FORM 40FR12B/A

Registration of a class of securities of certain Canadian issuers pursuant to Section 12(b) of the 1934 Act [amend]

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FILER

Kirkland Lake Gold Ltd.	Mailing Address	Business Address
CIK:**1713443**| IRS No.: **000000000** | State of Incorp.:**A6** | Fiscal Year End: **1231** Type: **40FR12B/A** | Act: **34** | File No.: **001-38179** | Film No.: **171009658**	*200 BAY STREET, SUITE 3120* *TORONTO A6 M5J 2J1*	*200 BAY STREET, SUITE 3120* *TORONTO A6 M5J 2J1* *416-840-7884*

UNITED STATES
SECURITIES AND EXCHANGE COMMISSION
Washington, D.C. 20549

FORM 40-F/A
(Amendment No. 2)

[X] Registration statement pursuant to Section 12 of the Securities Exchange Act of 1934

or

[ ] Annual report pursuant to Section 13(a) or 15(d) of the Securities Exchange Act of 1934

For the fiscal year ended ______________________ Commission File Number **001-38179**

Kirkland Lake Gold Ltd.
(Exact name of Registrant as specified in its charter)

Ontario	**1000**	**Not Applicable**
(Province or other jurisdiction of incorporation or organization)	(Primary Standard Industrial Classification Code Number)	(I.R.S. Employer Identification Number)

200 Bay Street, Suite 3120
Toronto, Ontario M5J 2J1
Canada
(416) 840-7884
(Address and telephone number of Registrant' s principal executive offices)

Registered Agent Solutions, Inc.
99 Washington Avenue
Suite 1008
Albany, NY 12260
(888) 705-7274
(Name, address (including zip code) and telephone number (including area code) of agent for service in the United States)

Securities registered or to be registered pursuant to Section 12(b) of the Act:

Title of each class	Name of each exchange on which registered
Common Shares, no par value	**New York Stock Exchange**

Securities registered pursuant to Section 12(g) of the Act: None.

Securities for which there is a reporting obligation pursuant to Section 15(d) of the Act: None

For annual reports, indicate by check mark the information filed with this Form:

[ ] Annual information form [ ]Audited annual financial statements

Indicate the number of outstanding shares of each of the registrant' s classes of capital or common stock as of the close of the period covered by the annual report: **N/A**

Indicate by check mark whether the registrant: (1) has filed all reports required to be filed by Section 13 or 15(d) of the Exchange Act during the preceding 12 months (or for such shorter period that the registrant was required to file such reports); and (2) has been subject to such filing requirements for the past 90 days. [ ] Yes [ ] No

Indicate by check mark by filing the information contained in this Form is also thereby furnishing the information to the Commission pursuant to Rule 12g3-2(b) under the Securities Exchange Act of 1934 (the "Exchange Act"). If "Yes" is marked, indicate the file number assigned to the Registrant in connection with such Rule. [ ] Yes [X] No

Indicate by check mark whether the registrant is an emerging growth company as defined in Rule 12b-2 of the Exchange Act.

Emerging growth company [X]

If an emerging growth company that prepares its financial statements in accordance with U.S. GAAP, indicate by check mark if the registrant has elected not to use the extended transition period for complying with any new or revised financial accounting standards† provided pursuant to Section 13(a) of the Exchange Act. [ ]

The term “new or revised financial accounting standard” refers to any update issued by the Financial Accounting Standards Board to its Accounting Standards Codification after April 5, 2012.

EXPLANATORY NOTE

Kirkland Lake Gold Ltd. (the "Company", the "Registrant") is a Canadian issuer eligible to file its registration statement pursuant to Section 12 of the Securities Exchange Act of 1934, as amended (the "Exchange Act"), on Form 40-F pursuant to the multi-jurisdictional disclosure system of the Exchange Act. The Company is a "foreign private issuer" as defined in Rule 3b-4 under the Exchange Act. Equity securities of the Company are accordingly exempt from Sections 14(a), 14(b), 14(c), 14(f) and 16 of the Exchange Act pursuant to Rule 3a12-3.

The Company filed a Registration Statement on Form 40-F on August 4, 2017 (the "Original Form 40-F") and an Amendment No. 1 to the Original Form 40-F on August 4, 2017 (the "Amendment No. 1). The Company is filing this Amendment No. 2 for the sole purpose of filing exhibits that were too large to be filed with the Original Form 40-F and the Amendment No. 1.

SIGNATURES

Pursuant to the requirements of the Exchange Act, the Registrant certifies that it meets all of the requirements for filing on Form 40-F and has duly caused this Registration Statement to be signed on its behalf by the undersigned, thereunto duly authorized.

KIRKLAND LAKE GOLD LTD.

By: /s/ Jennifer Wagner

Name: Jennifer Wagner

Title: Corporate Secretary

Date: August 4, 2017

EXHIBIT INDEX

The following documents are being filed with the Commission as Exhibits to this Registration Statement:

Exhibit	Description
99.1*	Annual Audited Consolidated Financial Statements for Kirkland Lake Gold Ltd. as at December 31, 2016, December 31, 2015, April 30, 2015 and April 30, 2014 and the Year Ended December 31, 2016, the Eight-Month Period Ended December 31, 2015 and Year Ended April 30, 2015*
99.2*	Management's Discussion and Analysis for the year ended December 31, 2016*
99.3*	Annual Information Form dated March 30, 2017*
99.4*	Certification of Refiled Annual Financial Statements by the CEO dated August 1 2017*
99.5*	Certification of Refiled Annual Financial Statements by the CFO dated August 1 2017*
99.6*	Indenture dated July 19, 2012*
99.7*	Supplemental Indenture dated November 7, 2012*
99.8*	Arrangement Agreement dated November 16, 2015*
99.9*	News Release dated January 11, 2016*
99.10*	News Release dated January 18, 2016*
99.11*	News Release dated January 26, 2016*
99.12*	Articles of Arrangement dated January 26, 2016*
99.13*	News Release dated February 2, 2016*
99.14*	News Release dated February 12, 2016*
99.15*	Material Change Report dated February 17, 2016*
99.16*	News Release dated February 26, 2016*
99.17*	News Release dated February 29, 2016*
99.18*	News Release dated March 4, 2016*
99.19*	Consolidated Financial Statements for the years ended December 31, 2015 and 2014*
99.20*	Management's Discussion and Analysis for the years ended December 31, 2015 and 2014*
99.21*	Confirmation of Notice of Record and Meeting Dates dated March 16, 2016*
99.22*	News Release dated March 21, 2016*
99.23**	Technical Report for the Maud Creek Gold Project, Northern Territory Australia dated March 21, 2016**
99.24**	Technical Report for the Stawell Gold Mine, Victoria, Australia dated March 16, 2016**
99.25**	Report on the Mineral Resources & Minerals Reserves of the Northern Territory Operations, Northern Territory, Australia dated March 21, 2016**
99.26	Report on the Mineral Resources & Mineral Reserves of the Fosterville Gold Mine, Victoria, Australia dated March 21, 2016

99.27*	Annual Information Form for the year ended December 31, 2015*
99.28*	Certification of Annual Filings in connection with filing of Annual Information Form by CEO March 21, 2016*
99.29*	Certification of Annual Filings in connection with filing of Annual Information Form by CFO March 21, 2016*

Exhibit	Description
99.30*	Material Change Report dated March 21, 2016*
99.31*	Revised Confirmation of Notice of Record and Meeting Dates dated March 28, 2016*
99.32*	News Release dated March 30, 2016*
99.33*	News Release dated April 4, 2016*
99.34*	News Release dated April 6, 2016*
99.35*	News Release dated April 12, 2016*
99.36*	Notice of Annual General Meeting of Shareholders dated April 7, 2016*
99.37*	Management Information Circular dated April 7, 2016*
99.38*	Form of Proxy dated April 22, 2016*
99.39*	News Release dated April 26, 2016*
99.40*	News Release dated April 29, 2016*
99.41*	Condensed Interim Consolidated Financial Statements for the three months ended March 31, 2016 and 2015*
99.42*	Management's Discussion and Analysis for the three months ended March 31, 2016 and 2015*
99.43*	Certification of Interim Filings by CEO April 29, 2016*
99.44*	Certification of Interim Filings by CFO April 29, 2016*
99.45*	News Release dated May 9, 2016*
99.46*	News Release dated May16, 2016*
99.47	Technical Report and Preliminary Economic Assessment of the Maud Creek Gold Project, Northern Territory, Australia dated May 16, 2016
99.48*	News Release dated May 18, 2016*
99.49	Amended Technical Report and Preliminary Economic Assessment of the Maud Creek Gold Project, Northern Territory, Australia dated May 18, 2016
99.50*	Material Change Report dated May 18, 2016*
99.51*	News Release dated May 26, 2016*
99.52*	Report of voting results dated May 26, 2016*
99.53*	News release dated June 27, 2016*
99.54*	News release dated July 12, 2016*
99.55*	News release dated July 29, 2016*
99.56*	Management's Discussion and Analysis for the three and six months ended June 30, 2016*

99.57*	Condensed Interim Consolidated Financial Statements for the three and six months ended June 30, 2016 and 2015*
99.58*	Certification of Interim Filings by CEO dated July 29, 2016,*
99.59*	Certification of Interim Filings by CFO dated July 29, 2016*
99.60*	News release dated August 3, 2016*
99.61*	News release dated August 22, 2016*
99.62*	News release dated September 14, 2016*

Exhibit	Description
99.63*	News release dated September 20, 2016*
99.64*	News release dated September 29, 2016*
99.65*	Form of Voting and Support Agreement dated September 29, 2016 re Kirkland Lake Gold Inc.*
99.66*	Form of Voting and Support Agreement dated September 29, 2016 re Kirkland Lake Gold Inc.*
99.67*	Form of Voting and Support Agreement dated September 29, 2016 re Newmarket Gold Inc.*
99.68*	Arrangement Agreement dated September 29, 2016*
99.69*	Material Change Report dated October 4, 2016*
99.70*	Confirmation of Notice of Record and Meeting Dates dated October 12, 2016*
99.71*	News release dated October 13, 2016*
99.72*	Revised Confirmation of Notice of Record and Meeting Dates dated October 13, 2016*
99.73*	Certificate of Officer dated October 31, 2016*
99.74*	Notice of Special Meeting of Shareholder of Newmarket Gold Inc. dated October 28, 2016*
99.75*	Joint Management Information Circular Concerning an Arrangement Involving Kirkland Lake Gold Inc. and Newmarket Gold Inc. dated October 28, 2016*
99.76*	Annual Information Form of Kirkland Lake Gold Inc. dated March 10, 2016*
99.77*	Audited Financial statements of Kirkland Lake Gold Inc. for the stub year ended December 31, 2015 and the year ended April 30, 2015*
99.78*	Management's Discussion and Analysis of Kirkland Lake Gold Inc. for the eight month (stub) year ended December 31, 2015*
99.79*	Unaudited Condensed Consolidated Interim Financial Statements of Kirkland Lake Gold Inc. as at and for the three and six month period ended June 30, 2016 and July 31, 2015*
99.80*	Management's Discussion and Analysis of Kirkland Lake Gold Inc. for the three and six months ended June 30, 2016*
99.81*	Management Information Circular of Kirkland Lake Gold Inc. dated May 16, 2016*
99.82*	Management Information Circular of Kirkland Lake Gold Inc. dated December 15, 2015*
99.83*	Management information circular of Kirkland Lake Gold Inc. dated September 23, 2015*
99.84*	Material Change Report of Kirkland Lake Gold Inc. dated October 3, 2016*
99.85*	Material Change Report of Kirkland Lake Gold Inc. dated January 27, 2016*
99.86*	News Release dated August 2, 2017*
99.87*	Joint Management Information Circular of Newmarket Gold Inc. and Crocodile Gold Corp. dated June 2, 2015*
99.88*	News Release dated October 31, 2016*

99.89*	Form of proxy dated October 31, 2016*
99.90*	News Release dated November 3, 2016*
99.91*	Management's Discussion and Analysis for the three and nine months ended September 30, 2016*
99.92*	Condensed Interim Consolidated Financial Statements for the three and nine months ended September 30, 2016 and 2015*
99.93*	Certification of Interim Filings by CEO dated November 3, 2016*

Exhibit	**Description**
99.94*	Certification of Interim Filings by CFO dated November 3, 2016*
99.95*	News Release dated November 8, 2016*
99.96*	News Release dated November 9, 2016*
99.97*	News Release dated November 11, 2016*
99.98*	News Release dated November 25, 2016*
99.99*	Report of voting results dated November 25, 2016*
99.100*	Letter of Transmittal for Registered Holders of Common Shares of Newmarket Gold Inc. dated November 29, 2016*
99.101*	News Release dated November 30, 2016*
99.102*	Articles of Amendment dated November 30, 2016*
99.103*	Notice of Change in Corporate Structure Pursuant to Section 4.9 of National Instrument 51-102 dated December 2, 2016*
99.104*	News Release dated November 30, 2016*
99.105*	Second Supplemental Indenture dated as of November 30, 2016*
99.106*	Material Change Report dated December 2, 2016*
99.107*	News Release dated December 6, 2016*
99.108*	News Release dated December 12, 2016*
99.109*	News Release dated December 23, 2016*
99.110*	News Release dated January 3, 2017*
99.111*	Report of Exempt Distribution dated January 3, 2017*
99.112*	News Release dated January 9, 2017*
99.113*	News Release dated January 17, 2017*
99.114*	News Release dated January 19, 2017*
99.115*	News Release dated January 30, 2017*
99.116*	News Release dated February 27, 2017*
99.117*	News Release dated March 6, 2017*
99.118*	Confirmation of Notice of Record and Meeting Dates dated March 10, 2017*
99.119*	News Release dated March 28, 2017*
99.120*	News Release dated March 29, 2017*
99.121*	Third Supplemental Indenture dated March 13, 2017*

99.122***	Report on the Mineral Resources & Mineral Reserves of the Northern Territory Operations, Northern Territory, Australia dated March 30, 2017***
99.123***	Macassa Property, Ontario, Canada Updated NI 43-101 Technical Report dated March 30, 2017***
99.124***	Holt-Holloway Property, Ontario, Canada Updated NI 43-101 Technical Report dated March 30, 2017***
99.125****	Report on the Mineral Resources & Mineral Reserves of the Stawell Gold Mine, Victoria, Australia dated March 30, 2017****
99.126****	Hislop Property, Ontario, Canada Updated NI 43-101 Technical Report dated March 30, 2017****

Exhibit	Description
99.127****	Report on the Mineral Resources & Minerals Reserves of the Fosterville Gold Mine, Victoria, Australia dated March 30, 2017****
99.128****	Taylor Property, Ontario, Canada Updated NI 43-101 Technical Report dated March 30, 2017****
99.129*	News Release dated March 30, 2017*
99.130*	Voting Instruction Form dated April 11, 2017*
99.131*	Notice of Annual General Meeting of Shareholders dated April 7, 2017*
99.132*	Management Information Circular dated April 7, 2017*
99.133*	Form of Proxy dated April 11, 2017*
99.134*	Kirkland Lake Gold Ltd. Long Term Incentive Plan dated April 7, 2017*
99.135*	Kirkland Lake Gold Ltd. Deferred Share Unit Plan dated April 7, 2017*
99.136*	Code of Conduct dated April 11, 2017*
99.137*	News Release dated April 12, 2017*
99.138*	Form of Proxy dated April 12, 2017*
99.139*	News Release dated April 24, 2017*
99.140*	News Release dated May 3, 2017*
99.141*	News Release dated May 4, 2017*
99.142*	Condensed Consolidated Interim Financial Statements for the three months ended March 31, 2017 and 2016*
99.143*	Management's Discussion and Analysis for the three months ended March 31, 2017 and 2016*
99.144*	Certification of Interim Filings by CEO dated May 4, 2017*
99.145*	Certification of Interim Filings by CFO dated May 4, 2017*
99.146*	Report of voting results dated May 4, 2017*
99.147*	News Release dated May 5, 2017*
99.148*	News Release dated May 15, 2017*
99.149*	News Release dated May 23, 2017*
99.150*	Annual Report 2016*
99.151*	News Release dated June 19, 2017*
99.152*	News Release dated June 21, 2017*
99.153*	News Release dated June 27, 2017*
99.154*	News Release dated June 28, 2017*

99.155*	News Release dated July 9, 2017*
99.156*	News Release dated July 27, 2017*
99.157*	Condensed Consolidated Interim Financial Statements for the three and six months ended June 30, 2017 and 2016*
99.158*	Management’s Discussion and Analysis for the three and six months ended June 30, 2017 and 2016*
99.159*	Certification of Interim Filings by CEO dated August 1, 2017*
99.160*	Certification of Interim Filings by CFO dated August 1, 2017*

Exhibit	**Description**
99.161*	Consent of Jason Keily*
99.162*	Consent of Peter Fairfield*
99.163*	Consent of SRK Consulting (Australia) Pty Ltd.*
99.164*	Consent of David Schonfeldt*
99.165*	Consent of Danny Kentwell*
99.166*	Consent of Justine Tracey*
99.167*	Consent of Mark Edwards*
99.168*	Consent of Wayne Chapman*
99.169*	Consent of Murray Smith*
99.170*	Consent of Troy Fuller*
99.171*	Consent of Ion Hann*
99.172*	Consent of Mining Plus PTY Ltd.*
99.173*	Consent of Simon Walsh*
99.174*	Consent of Pierre Rocque*
99.175*	Consent of Douglas Carter*
99.176*	Consent of John Winterbottom*
99.177*	Consent of Ian Holland*
99.178*	Consent of Glenn R. Clark*
99.179*	Consent of Glenn R. Clark & Associates*
99.180*	Consent of Stewart Carmichael*
99.181*	Consent of Christopher Stewart*
99.182*	Consent of Keyvan Salehi*
99.183*	Consent of Dean Basile*
99.184*	Consent of Phil Bremner*
99.185*	Consent MiningOne Pty*
99.186*	Consent of Simon Hitchman*
99.187*	Consent of Stuart Hutchin*
99.188*	Consent of GMP Securities L.P.*

99.189*	Consent of CIBC World Markets Inc.*
99.190*	Consent of RBC Dominion Securities Inc.*
99.191*	Consent of Maxit Capital LP*
99.192*	Consent of PricewaterhouseCoopers LLP*
99.193*	Consent of KPMG LLP*

* previously filed with the Original Form 40-F
** previously filed with Amendment No. 1
*** to be filed with Amendment No. 3 to this Registration Statement on Form 40-F
**** to be filed with Amendment No. 4 to this Registration Statement on Form 40-F

REPORT ON THE
MINERAL RESOURCES & MINERAL RESERVES
OF THE FOSTERVILLE GOLD MINE
VICTORIA, AUSTRALIA
EFFECTIVE DATE DECEMBER 31, 2015
Dated March 21, 2016

PREPARED FOR

Newmarket Gold

Troy Fuller, BSc Hons MAIG Fosterville Gold Mine Geology Manager Newmarket Gold	Ion Hann, BEng (Mining) FAusIMM Fosterville Gold Mine Mining Manager Newmarket Gold

REPORT ON THE MINERAL RESOURCES & MINERAL RESERVES OF THE FOSTERVILLE GOLD MINE, VICTORIA, AUSTRALIA. Effective Date: DECEMBER 31, 2015

Prepared For Newmarket Gold Inc.

Ion Hann (Mining Manager – Fosterville Gold Mine), BEng (Mining), FAusIMM
Troy Fuller (Geology Manager – Fosterville Gold Mine), BSc (Geology) Hons, MAIG

IMPORTANT NOTICE

This report has been prepared as a National Instrument 43-101 Technical Report, as prescribed in Canadian Securities Administrators' National Instrument 43-101 (NI 43-101) for Newmarket Gold Inc. (Newmarket Gold): The data, information, estimates, conclusions and recommendations contained herein, as prepared and presented by the Authors, are consistent with: i) information available at the time of preparation; ii) data supplied by outside sources, which has been verified by the authors, as applicable; and, iii) the assumptions, conditions and qualifications set forth in this report.

CAUTIONARY NOTE WITH RESPECT TO FORWARD LOOKING INFORMATION

This document contains "forward-looking information" as defined in applicable securities laws. Forward looking information includes, but is not limited to, statements with respect to the future production, costs and expenses of the project; the other economic parameters of the project, as set out in this technical report, including; the success and continuation of exploration activities, including drilling; estimates of mineral reserves and mineral resources; the future price of gold; government regulations and permitting timelines; requirements for additional capital; environmental risks; and general business and economic conditions. Often, but not always, forward-looking information can be identified by the use of words such as "plans", "expects", "is expected", "budget", "scheduled", "estimates", "continues", "forecasts", "projects", "predicts", "intends", "anticipates" or "believes", or variations of, or the negatives of, such words and phrases, or statements that certain actions, events or results "may", "could", "would", "should", "might" or "will" be taken, occur or be achieved. Forward-looking information involves known and unknown risks, uncertainties and other factors which may cause the actual results, performance or achievements to be materially different from any of the future results, performance or achievements expressed or implied by the forward-looking information. These risks, uncertainties and other factors include, but are not limited to, the assumptions underlying the production estimates not being realized, decrease of future gold prices, cost of labour, supplies, fuel and equipment rising, the availability of financing on attractive terms, actual results of current exploration, changes in project parameters, exchange rate fluctuations, delays and costs inherent to consulting and accommodating rights of local communities, title risks, regulatory risks and uncertainties with respect to obtaining necessary permits or delays in obtaining same, and other risks involved in the gold production, development and exploration industry, as well as those risk factors discussed in Newmarket Gold Inc.' s latest Annual Information Form and its other SEDAR filings from time to time. Forward-looking information is based on a number of assumptions which may prove to be incorrect, including, but not limited to, the availability of financing for Newmarket Gold Inc.' s production, development and exploration activities; the timelines for Newmarket Gold Inc.' s exploration and development activities on the property; the availability of certain consumables and services; assumptions made in mineral resource and mineral reserve estimates, including geological interpretation grade, recovery rates, price assumption, and operational costs; and general business and economic conditions. All forward-looking information herein is qualified by this cautionary statement. Accordingly, readers should not place undue reliance on forward-looking information. Newmarket Gold Inc. and the authors of this technical report undertake no obligation to update publicly or otherwise revise any forward-looking information whether as a result of new information or future events or otherwise, except as may be required by applicable law.

NON-IFRS MEASURES

This technical report contains certain non-International Financial Reporting Standards measures. Such measures have non standardized meaning under International Financial Reporting Standards and may not be comparable to similar measures used by other issuers.

FIGURES		PAGE NO.

1 EXECUTIVE SUMMARY

This document has been prepared for Newmarket Gold Inc. (Newmarket Gold), the beneficial owner of the Fosterville Gold Mine. Newmarket Gold is listed on the Toronto Stock Exchange (NMI). This document provides the Mineral Resource and Mineral Reserve estimates for the Fosterville Gold Mine that have resulted from ongoing exploration and resource definition drilling and as a result of ongoing mine design and evaluation during the period from December 31, 2014 to December 31, 2015.

Location

The Fosterville Gold Mine (Fosterville or FGM) is located approximately 20km north-east of the city of Bendigo and 130km north of the city of Melbourne in Victoria, Australia.

History and Ownership

Gold was first discovered in the Fosterville area in 1894 with mining activity continuing until 1903 for a total of 28,000 ounces of production. Mining in this era was confined to the near-surface oxide material. Aside from a minor tailings retreatment in the 1930' s, activity resumed in 1988 with a further tailings retreatment program conducted by Bendigo Gold Associates which ceased in 1989. Mining recommenced in 1991 when Brunswick Mining NL (Brunswick) and then Perseverance Corporation Ltd. (Perseverance or PSV) (from 1992) commenced heap leaching operations from shallow oxide open pits. Between 1988 and the cessation of oxide mining in 2001, a total of 240,000 ounces of gold were poured (Roberts *et al*, 2003).

A feasibility study into a sulphide mining operation was completed by Perseverance in 2003 with construction and open pit mining commencing in early 2004. Commercial production commenced in April 2005 and up to the end of December 2006 had produced 136,882 ounces of gold. In October 2007, Perseverance announced that it had entered into an agreement with Northgate Minerals Corporation (Northgate) to acquire the company. Full control passed to Northgate in February 2008. The 500,000th ounce of sulphide gold production was subsequently achieved in April 2011.

In August 2011, Northgate entered into a merger agreement with AuRico Gold Inc. (AuRico), who assumed control of Northgate in October 2011. However, in March 2012 AuRico and Crocodile Gold Corp (Crocodile Gold) jointly announced that Crocodile Gold would acquire the Fosterville and Stawell Mines. Crocodile Gold' s ownership of Fosterville was achieved on May 4, 2012. In early July 2015, Newmarket Gold Inc. merged with Crocodile Gold to form Newmarket Gold Inc.

Geology and Mineralization

The Fosterville Goldfield is located within the Bendigo Structural Zone in the Lachlan Fold Belt. The deposit is hosted by an interbedded turbidite sequence of sandstones, siltstones and shales. This sequence has been metamorphosed to sub-greenschist facies and folded into a set of upright, open to closed folds. The folding resulted in the formation of a series of bedding parallel laminated quartz (LQ) veins. Although visually similar to their mineralized equivalents at Bendigo (20km away), these LQ veins at Fosterville are effectively unmineralized.

Mineralization at Fosterville is controlled by late brittle faulting. These late brittle faults are generally steeply west dipping reverse faults with a series of moderately west dipping reverse splay faults formed in the footwall of the main fault. There are also moderately east dipping faults which have become more significant footwall to the anticlinal offsets along the west dipping faults. Primary gold mineralization occurs as disseminated arsenopyrite and pyrite forming as a selvage to veins in a quartz-carbonate veinlet stockwork. The mineralization is structurally controlled with high-grade zones localized by the geometric relationship between bedding and faulting. Mineralized shoots are typically 4m to 15m thick, 50m to 150m up/down dip and 300m to 1,500m+ down plunge.

Antimony mineralization, in the form of stibnite, occurs with quartz and varies from replacement and infill of earlier quartz-carbonate stockwork veins, to massive stibnite-only veins of up to 0.5m in width. The stibnite-quartz event occurs in favorable structural locations, such as the Phoenix Eagle and Lower Phoenix structures. There are also occurrences of primary visible gold (≤3mm in size) that have a spatial association with stibnite in fault related quartz veins. The occurrence of visible gold is becoming increasingly significant and is being observed more frequently with depth and down plunge within the Lower Phoenix Mineralized Zones.

Crocodile Gold engaged Quantitative Group (QG) in November 2014, in response to the noted increased frequency of visible gold occurrences at depth, to provide FGM with some external advice and thinking regarding the implications to resource estimation and mine geology practices. In 2015, QG continued to assist FGM through review of current practices and providing technical theory and background to sampling, assaying and resource modeling in visible gold environments.

Current Status

Since the commencement of commercial gold production in April 2005, the sulphide plant at Fosterville Gold Mine has produced 1,000,682 ounces of gold up to the end of December 2015. This production was initially sourced solely from open cut mining with underground mining starting to contribute from late 2006. The Harrier open cut was initially completed in December 2007 and since that time the underground mine has been the primary source of ore. Ore sourced from a series of pit expansions on the previously mined Harrier, John' s and O' Dwyer's South Pits between Q1 2011 and Q4 2012 has provided supplementary feed to underground ore sources. Since the beginning of 2013 underground operations have been the sole provider of mill feed at Fosterville. Current mining activities are focused on the Central, Phoenix and Harrier underground areas and current gold production outlook for 2016 is between 110,000 - 120,000 ounces. Newmarket Gold is also planning to undertake 110,624m of exploration and resource definition drilling and continue the development of a dedicated underground drill platform. Total estimated cost for exploration and resource development activities for 2016 is AUD$22M.

Mineral Resources and Mineral Reserves

The Mineral Resources and Mineral Reserves reported are broken down into areas contained within the mine lease MIN5404 (Section 4). Mineral Resource Areas of Central, Southern, Harrier and Robbins Hill (Table 1-2) are historically defined resource areas which were established at different times in the evolution of the project. The Central and Robbins Hill Areas contain multiple mineral resource models primarily for reasons of data handling. Details on mineral resource block model extents can be seen in Figure 14-1.

All Mineral Reserves are contained within the Central and Harrier Mineral Resource Areas. Mineral Reserves contained within the Central Mineral Resource Area have been subdivided into Central, Phoenix and Eagle Mineral Reserves Table 1-3

CIL Residue Mineral Resource and Mineral Reserves are distinguished from insitu Mineral Resources and Mineral Reserves in Table 1-1, Table 1-2 & Table 1-3 on the basis of differing recovery assumptions.

Table 1-1 Summarized Mineral Resources (Inclusive of Mineral Reserve) for FGM as at December 31, 2015.

Summarized Mineral Resources (inclusive of Mineral Reserve) for Fosterville as of December 31, 2015			
Classification	**Tonnes (kt)**	**Gold Grade (g/t Au)**	**Insitu Gold (kOz)**
Oxide and Sulphide Materials			
Measured	2,086	3.25	218
Indicated	12,950	4.57	1,904
Total (Measured and Indicated)	**15,036**	**4.39**	**2,122**
Inferred	**5,073**	**4.08**	**665**

Classification	Tonnes (kt)	Gold Grade (g/t Au)	Insitu Gold (kOz)
CIL Residues			
Measured	**571**	**7.83**	**144**

Notes:

For the Mineral Resource estimate, the Qualified Person is Troy Fuller.
The Mineral Resources reported are inclusive of the Mineral Reserves.
See notes provided for Table 1-2 for more detail on oxide and sulphide resources.
CIL residues are stated as contained ounces – 25% recovery is expected. Recoveries are based on operating performances
Mineral Resources are rounded to 1,000 tonnes, 0.01 g/t Au and 1,000 ounces. Minor discrepancies in summation may occur due to rounding.
Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
The Mineral Resource estimate used a gold price of AUD$1500 per ounce.

Table 1-2 Mineral Resources (Inclusive of Mineral Reserve) for FGM as at December 31, 2015.

Mineral Resources (Inclusive of Mineral Reserves) - Fosterville as at December 31, 2015										
Classification		Measured			Indicated			Inferred		
		Tonnes (kt)	Grade (g/t Au)	Insitu Gold (kOz)	Tonnes (kt)	Grade (g/t Au)	Insitu Gold (kOz)	Tonnes (kt)	Grade (g/t Au)	Insitu Gold (kOz)
Fosterville Fault Zone Sulphide Resources										
Central Area	Upper	1,463	2.47	116	808	2.69	70	24	1.45	1
	Lower	315	8.29	84	5,188	6.65	1,109	1,488	5.58	267
Southern Area	Upper	21	3.32	2	463	2.44	36	537	2.29	40
	Lower	0	0.00	0	0	0.00	0	320	5.59	57
Harrier Area	Upper	0	0.00	0	0	0.00	0	0	0.00	0
	Lower	17	5.03	3	2,720	5.65	494	1,266	5.64	230
Robbin's Hill Area Sulphide Resources										
Combined	Upper	0	0.00	0	1,787	1.77	102	976	1.51	47
	Lower	0	0.00	0	114	3.81	14	59	3.38	6
Sulphide Upper		1,484	2.48	118	3,058	2.11	208	1,537	1.78	88
Sulphide Lower		332	8.12	87	8,022	6.27	1,617	3,132	5.57	560
Total Sulphide		**1,816**	**3.51**	**205**	**11,080**	**5.12**	**1,825**	**4,669**	**4.32**	**648**
Total Oxide		**270**	**1.47**	**13**	**1,870**	**1.32**	**80**	**404**	**1.27**	**16**
Total Oxide & Sulphide		**2,086**	**3.25**	**218**	**12,950**	**4.57**	**1,904**	**5,073**	**4.08**	**665**

Classification	Measured			Indicated			Inferred		
	Tonnes (kt)	Grade (g/t Au)	Insitu Gold (kOz)	Tonnes (kt)	Grade (g/t Au)	Insitu Gold (kOz)	Tonnes (kt)	Grade (g/t Au)	Insitu Gold (kOz)
Residues									
CIL	**571**	**7.83**	**144**	**0**	**0.00**	**0**	**0**	**0.00**	**0**
Total	**571**	**7.83**	**144**	**0**	**0.00**	**0**	**0**	**0.00**	**0**

Notes:

For the Mineral Resource estimate, the Qualified Person is Troy Fuller.

The Mineral Resources reported are inclusive of the Mineral Reserves for the same area.

Lower cut-off grades applied to Mineral Resources are 0.7 g/t Au for oxide and 1.0 g/t Au for sulphide mineralization above 5050mRL (approximately 100m below surface), which is deemed to be potentially open-pitable. A lower cut-off grade of 3.0 g/t Au is applied to Mineral Resource material below 5050mRL.

CIL residue is stated as contained ounces – 25% recovery is expected. Recoveries are based on operating performances.
Mineral Resources are rounded to 1,000 tonnes, 0.01 g/t Au and 1,000 ounces. Minor discrepancies in summation may occur due to rounding.
Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
The Mineral Resource estimate used a gold price of AUD$1500 per ounce.

Table 1-3 Mineral Reserves for FGM as at December 31, 2015.

Mineral Reserves - Fosterville as at December 31, 2015									
	Proven			Probable			Total		
Classification	Tonnes (kt)	Grade (g/t Au)	In situ Gold (kOz)	Tonnes (kt)	Grade (g/t Au)	In situ Gold (kOz)	Tonnes (kt)	Grade (g/t Au)	In situ Gold (kOz)
Underground									
Central	72	5.11	12	149	4.58	22	220	4.75	34
Phoenix	120	5.24	20	531	8.20	140	651	7.65	160
Eagle	24	8.24	6	149	8.00	38	173	8.03	45
Harrier	17	3.69	2	30	3.32	3	47	3.45	5
Surface									
	0	0.00	0	0	0.00	0	0	0.00	0
Total	**232**	**5.39**	**40**	**859**	**7.36**	**203**	**1,091**	**6.95**	**244**

	Proven			Probable			Total		
Classification	Tonnes (kt)	Grade (g/t Au)	In situ Gold (kOz)	Tonnes (kt)	Grade (g/t Au)	In situ Gold (kOz)	Tonnes (kt)	Grade (g/t Au)	In situ Gold (kOz)
Residues									
CIL	571	7.83	144	0	0.00	0	571	7.83	144
Total	**571**	**7.83**	**144**	**0**	**0.00**	**0**	**571**	**7.83**	**144**

Notes:

For the Mineral Reserves estimate, the Qualified Person is Ion Hann. The Mineral Reserve estimate used a gold price of AUD$1,450 per ounce. The lower cut-off grades applied ranged from 1.6 g/t to 2.7 g/t Au for underground sulphide ore depending upon width, mining method and ground conditions.

Dilution of 20% and mining recovery of 80% were applied to stopes within the Mineral Reserves estimate.

Mineral Reserves are rounded to 1,000 tonnes, 0.01 g/t Au and 1,000 ounces. Minor discrepancies in summation may occur due to rounding.

CIL residue is stated as contained ounces – 25% recovery is expected. Recoveries are based on operating performances.

Conclusions and Recommendations

The authors have made the following interpretations and conclusions:

The understanding of the fundamental geological controls on mineralization at Fosterville is high. Primary mineralization is structurally controlled with high-grade zones localized by the geometric relationship between bedding and west dipping faulting. This predictive model has led to considerable exploration success in following the down-plunge extensions of high -grade mineralization.

- The **Lower Phoenix Fault** is the primary west dipping structure in the active mine development area and is defined by reverse faulting on a shale package where anticline thrust displacement of ~80 meters occurs. The fault dips between 35 and ~55 degrees to the west and mineralization can be traced along a dip extent of ~190m and strike extent of ~1. 3km. The dominate mineralization style on this structure is disseminated sulphide, however occurrences of visible gold at depth are becoming increasingly common, concentrated where footwall structures intersect. The Lower Phoenix System currently remains open to the north and south so maximum plunge extent has not yet been defined.

- Throughout 2015, development mapping and continued drilling confirmed that there were multiple mineralized structures of various size and continuity footwall to the main west dipping Lower Phoenix Fault, which present significant resource growth potential. Progressive geological understanding of the Phoenix and Lower Phoenix footwall environs has highlighted the significance of these favorable settings for mineralization, including;

    East Dipping mineralized structures, namely the **Eagle Fault** and **East Dipping Faults** which commonly contain quartz- stibnite vein assemblages and substantial concentrations of visible gold typically enveloped by halos of disseminated sulphide. The Eagle Fault is discordant to bedding and variably dips between 10 and 60 degrees to the east, where East Dipping Faults are typically bedding parallel to sub parallel with dips of ~70 degrees east to sub vertical.

    Low- angled **Lower Phoenix Footwall** west dipping structures typically consist of large quartz veins up to several meters wide with laminated textures, indicating a series of multiple mineralizing events, including a later stage quartz-stibnite phase of mineralization and visible gold. The faults are interpreted to have minimal offset but rather have been hydraulically fractured. Where these structures form linkages between the Lower Phoenix and East Dipping Faults, extremely high gold grades are observed.

    Continued drill definition of these structures over 2015, in combination with ore development and production exposure and reconciliation performance has reaffirmed the significance of these easterly dipping footwall structures to the Lower Phoenix Fault. The defined continuity, proximity to existing Mineral Resources and high grade tenor of these structures enhances the December 2015 Mineral Resource and Reserve position. Furthermore, mineralization on these structures is open down plunge, providing encouraging future Mineral Resource and Mineral Reserve growth potential for the operation.

    There is an observed change in the nature of some of Fosterville mineralization at depth with a number of high-grade, quartz-stibnite hosted, visible gold drill intercepts recorded for the Eagle, Lower Phoenix, Lower Phoenix Footwall and East Dipping Zones. Disseminated sulphide mineralization continues to persist at all depths and is uniform in character. It is currently inferred that the quartz-stibnite- visible gold assemblages have been emplaced at a later date to the disseminated sulphide providing an upgrade to the mineralization.

In addition, a better understanding of the mineralization of the **Kestrel System** was established during 2015. Drilling has defined an extensive broad zone of low to moderate grade disseminated sulphide mineralization (~average 6m in width) centered around the syncline hinge axial plane.

Progressive geological interpretation has led to continued development of robust geological and resource models underpinning the Mineral Resource and Mineral Reserve estimates. The relationship between mineralization and the controlling structural/stratigraphic architecture means that quality geological interpretation is critical to producing quality resource/reserve estimates.

The modifying factors used to convert the Mineral Resources to Mineral Reserves have been refined with the operating experience gained since underground production commenced in September 2006. In particular, the robustness of the mining recovery and dilution estimates has improved with experience relative to the pre- mining assessments.

The following recommendations are made:

Further mine lease growth exploration activities should be pursued. Given the strong understanding of geological controls on mineralization, this could have the potential to yield additional resources and reserves. Particular areas that are recommended to focus upon are the up and down -plunge extensions of the Lower Phoenix structure (northwards up-plunge from 7750mN and southwards down-plunge from 6200mN) Exploration of the Lower Phoenix southwards of 6200mN is technically challenging from surface due to target depths and as such Newmarket Gold has commenced the development of dedicated underground drill platform to facilitate further exploration of the Lower Phoenix system down plunge. The current 2016 exploration budget includes 320m of development from the Harrier decline to establish a hanging wall drilling platform to target Lower Phoenix extensions at a cost of AUD$2.95M and a combined total of 14,000m of diamond drilling for an estimated AUD$3.35M to explore these gold targets.

The infill/resource definition programs should be continued with an aim to maintain at least 12 months of reserves drilled out to 25m centers (or closer where necessary). Both the south plunging, westerly dipping Phoenix and Lower Phoenix Mineralized Zones and the easterly dipping Eagle and East Dipping Mineralized Zones require definition drilling which is to be conducted from both hangingwall (western side) and footwall (eastern side) drill platforms. The current infill drilling budget for 2016 includes 64,416m of drilling at an estimated cost of AUD$10.43M. As the decline and mining front continues to move south and lower, further hangingwall drives will be required to be developed. This work and the associated drilling have not been costed in detail.

It is also recommended that infill / resource definition programs target down plunge extensions of the Harrier Mineralized Zones with the aim to increase Mineral Reserves. Aspirational reserve work on Inferred Mineral Resources in the Harrier South area indicated that with smaller scale mining parameters applied these Mineral Resources have the potential to be converted into Mineral Reserves that would have high potential to increase the current LOM at FGM. Newmarket Gold has incorporated into its 2016 budget planned drilling into the Harrier South area with the aim to increase mineral resource confidence to allow for mineral reserve evaluation. This budgeted phased drill program consists of 32,208m at an estimated cost of AUD$5.21M.

The observed increased frequency of visible gold intercepts at depth requires continued research to better understand the potential implications on future geological, mining and metallurgical processes. Newmarket Gold continued to seek external advice over 2015 in relation to sampling, assaying and resource estimation of visible gold mineralization. Based on recommendations from external reviews, projects plans have been developed and implemented.

With this additional drilling data and further ongoing operational experience, it is recommended that mining recovery and dilution factors are reviewed and refined on an ongoing basis.

The proposed exploration activities in the current year are focused on targets close to Fosterville. However, in subsequent years, exploration drilling is planned for target areas away from the Fosterville mining lease within the north-south extensive EL3539. The intent of the exploration is to replace mineral reserves at Fosterville by extending presently known ore shoots, but to also locate and define new centers of economic gold mineralization applicable to open pit or underground extraction.

2 INTRODUCTION

2.1 Terms of Reference

The purpose of this technical report on Fosterville is to support public disclosure of Mineral Resource and Mineral Reserve estimates effective at Fosterville as at December 31, 2015. This report has been prepared in compliance with disclosure and reporting requirements set forth in the National Instrument 43-101 (NI 43-101) 'Standards of Disclosure for Mineral Projects' and Form 43-101F1.

This technical report has been prepared for Newmarket Gold, the beneficial owner of Fosterville. Newmarket Gold is listed on the Toronto Stock Exchange. Newmarket Gold is a Canadian-listed gold mining and exploration company with three operating mines in Australia, the Fosterville and Stawell Gold Mines in the state of Victoria and Cosmo Gold Mine in the Northern Territory. This technical report has been prepared for the use of Newmarket Gold to provide technical information to assist with business decisions and future project planning.

The report provides an update of the Mineral Resource and Mineral Reserve position as of December 31, 2015. The Mineral Resources and Mineral Reserve estimate for Fosterville is a summation of a number of individual estimates for various mineralized zones or various geographically constrained areas. All of these estimates are contained within the Mining Lease MIN5404 (MIN5404 or Fosterville Mine Lease). Details of the locations and geographical constraints of the various mineralized zones as of December 2015 are given in Section 14.

Gold was first discovered in the Fosterville area in 1894, aside from a minor tailings retreatment in the 1930' s, the field lay dormant until 1988 when Bendigo Gold Associates again recommenced gold production at Fosterville from the reprocessing of tailings. Mining operations and various levels of exploration and resource development activities have been continuous since 1988 and as such, the project has a significant past production and development history which is discussed in this report and also utilized during the compilation of the Mineral Resource and Mineral Reserve estimates.

The report includes an overview of Fosterville Gold Mine which has been compiled from company technical reports, published geological papers and internal Mineral Resource and Mineral Reserve documents completed by members of the FGM mine geological and mine engineering teams. The overview includes a description of the geology, project history, exploration activities and results, methodology, quality assurance, interpretations, metallurgy, land issues and environmental information. It also provides recommendations on additional exploration drilling which has the potential to upgrade resource classifications and to augment the resource base.

Mr. Troy Fuller of Fosterville is a Qualified Person as defined by NI 43-101 and accepts overall responsibility for the preparation of sections 1-14, 17, 18.2 – 27 and 28.2 of this report.

Mr. Ion Hann of Fosterville is a Qualified Person as defined by NI 43-101 and accepts overall responsibility for the preparation of sections 15-16, 18.1 and 28.1 of this report.

All information presented in this report was prepared in accordance with the requirements of National Instrument 43-101, Standards of Disclosure for Mineral Projects and is in the format prescribed by that instrument.

2.2 Field Involvement of Qualified Persons

Ion Hann is the Mining Manager for FGM. Ion has over 25 years of experience in the mining industry. In this time, 10 years of relevant experience in gold mining operations has been gained at Fosterville. Ion Hann is based at Fosterville.

Troy Fuller is the Geology Manager for FGM. Troy has over 20 years mining experience and has 17 years of gold operations experience in the Northern Territory, Western Australia and Victoria. Troy Fuller has managed all aspects of the geological operations for Fosterville since May 2010. Troy Fuller is based at Fosterville.

All of the Qualified Persons are based at Fosterville and through routine personal inspection have a comprehensive understanding of the property conditions, geology and mineralization, work completed and works planned /recommended.

2.3 Definitions

Table 2-1 Definition of Terms

Term	Description
ºC	Degrees Celsius
AAS	Atomic Absorption Spectroscopy
AC	Air core
acQuire	acQuire - Geoscientific Information Management System
Ag	Silver
AHD	Australian Height Datum (mean sea level)
AIG	Australian Institute of Geoscientists
Al	Aluminum
Aminya	Aminya Laboratory Services
Ammtec	ALS Ammtec Ltd.
ALS	Australian Laboratory Services
AMDEL	Amdel Analytical Laboratories
As	Arsenic
Au	Gold
AUD	Australian Dollars
AuRico	AuRico Gold Corporation
AusIMM	Australian Institute of Mining and Metallurgy
Ba	Barium
Bendigo Gold Associates	Bendigo Gold Associates Ltd., owner of the FGM prior to Brunswick
BETS-SHTS	Bendigo to Shepparton power line
BHP	Broken Hill Proprietary, now BHP Billiton
Bactech	Bactech (Australia) Ltd
Be	Beryllium
Bi	Bismuth
Biomin	Biomin South Africa Pty Limited
BIOX®	Proprietary bacterial oxidation technology licenced from Goldfields Ltd.
Brunswick	Brunswick Mining N.L., owner of the FGM prior to Perseverance
Ca	Calcium
CCD	Counter Current Decantation

Term	Description
Cd	Cadmium
Ce	Cerium
CIL	Carbon in Leach
CIL Residue	Carbon in Leach Residue. The term is equivalent to CIL Tailings.
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
cm	Centimetre
Co	Cobalt
CO3	Carbonate
COG	Cut-off Grade
CP	Chartered Professional
Cr	Chromium
CRM	Central Resource Model
Crocodile Gold	Crocodile Gold Corporation
Cu	Copper
DSDBI	Department of State Development, Business and Innovation
DTM	Digital Terrain Model
E	Easting
EL	Exploration Licence
EMS	Electronic Multi-shot Survey
EPA	Environment Protection Authority
ETW	Estimated True Width
Fe	Iron
FGM	Fosterville Gold Mine
FVTS	Fosterville Terminal Station
FW	Footwall
FY	Future Year
g/cm^3	Gram per cubic centimetre (unit of density)
GAL	Gekko Analytical Laboratories
GDA94	Geocentric Datum of Australia, 1994
GFL	Goldfields Limited
GC	Grade Control
GSV	Geoscience Victoria
g/t	Grams per (metric) tonne
HCl	Hydrogen Chloride
HDPE	High Density Polyethylene
HF	Hydrogen Fluoride
HG	High-grade

Historic Resource	A qualified person has not done sufficient work to classify historical estimates as current Mineral Resources or Mineral Reserves described within the report. Newmarket Gold is not treating any historical estimates as current Mineral Resources or Mineral Reserves.
HNO3	Nitric Acid
HQ	63.5 mm diameter diamond drill core
HRM	Harrier Resource Model
HW	Hangingwall
ICP-AES	Inductively Coupled Plasma - Atomic Emission Spectrometry
IP	Induced Polarization - geophysical imaging technique
ISO	International Organization for Standardization
JORC Code	Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (prepared by the Joint Ore Reserves Committee).

Term	Description
K	Potassium
km	Kilometre
km^2	Square kilometre (area)
koz	Kiloounce
kt	Kilotonne
K/Th	Potassium/Thorium ratio - relating to a 2008 airborne radiometric survey
kV	Kilovolt
kVA	Kilovolt-ampere
kW	Kilowatt
LG	Low-grade
LOM	Life of Mine
LQ	Laminated Quartz
LTK60	43.9 mm diameter diamond drill core
m	Metre
Ma	Million years
MCC	Motor Control Centre
MI	Metallurgy International
MIN	Mining Lease
Mg	Magnesium
MGA	Map Grid of Australia
µm	Micrometre
ML	Megalitre
mm	Millimetre
Mn	Manganese
Mo	Molybdenum
MVA	Megavolt-ampere
Mt	Million (metric) tonnes
Mtpa	Million (metric) tonnes per annum
N	Northing
Na	Sodium
NATA	National Association of Testing Authorities
Nb	Niobium
NCC	Non-carbonate carbon
New Holland	New Holland Mining Ltd., now Nu Energy Capital Limited
Newmarket	Newmarket Gold Inc.
Ni	Nickel
NI43-101	National Instrument 43-101

NNE	North North-East
NNW	North North-West
NRM	Northern Resource Model
Northgate	Northgate Minerals Corporation
NQ	47.6 mm diameter diamond drill core
NQ2	50.6 mm diameter diamond drill core
NW	Northwest
ODW	O' Dwyer's
ONAF	Oil Natural Air Forced – Transformer cooling without pumps and fans for air
ONAN	Oil Natural Air Natural - Transformer cooling without pumps and fans
O/O	Oblique /Oblique (structural setting)
O/P	Oblique /Parallel (structural setting)

Term	Description
OSLS	On Site Laboratory Services
oz	Troy Ounce (31.1034768 grams)
P	Phosphorous
PAF	Potentially Acid Forming
Pb	Lead
P/O	Parallel /Oblique (structural setting)
P/P	Parallel /Parallel (structural setting)
ppb	Parts per billion
PRM	Phoenix Resource Model
PQ	85.0 mm diameter diamond drill core
PSV	Perseverance Corporation Ltd., a wholly owned subsidiary of Newmarket Gold
Pt	Platinum
QAQC	Quality Assurance – Quality Control
QG	Quantitative Group (Geostatistical Consultants)
QP	Qualified Person
R^2	R squared – coefficient of determination
RAB	Rotary Air Blast
RC	Reverse Circulation
RH	Robbins Hill
Riffle splitter	A device comprising tiers of 'riffles' for equi-probable splitting of dry particulate matter (e.g. drill chips), each tier yields a 50:50 split.
RL	Reduced Level (elevation)
ROM	Run of Mine
RQD	Rock Quality Designation
S	Sulphur
SAG	Semi-Autogenous Grinding
Sb	Antimony – present at Fosterville in the mineral stibnite
SD	(Statistical) Standard Deviation
SMU	Selective Mining Unit
Sn	Tin
SP Ausnet	SP Ausnet – Electricity Distributor
Spear sampling	Using a tube ('spear') to collect a sample for assay from a sample bag of RC or RAB drill chips (this method is not equi-probable as it is susceptible to density segregation in the sample bag)
SQL	Structured Query Language
Sr	Strontium
t	(Metric) tonne (2204.6 lb or 1.1023 short tons)
Ta	Tantalum
Tailings	Ground rock and process effluents generated during processing of ore

TGC	Total Graphitic Carbon
Th	Thorium
Ti	Titanium
t/m^3	Tonne per cubic metre (unit of density)
TOEC	Total Organic and Elemental Carbon
Tl	Thallium
tpa	Tonnes Per Annum
TSF	Tailings Storage Facility
UG	Underground
V	Vanadium
W	Tungsten

Term	Description
WPV	Work Plan Variation
Zr	Zirconium
Y	Yttrium

2.4 Grids

Note that all Eastings, Northings, elevations (RL) and azimuths in the text reference to the local FGM grid. The FGM grid is a plane affine grid and can be referenced to MGA using the two reference points contained in Table 2-2 and -5000mRL (AHD). Fosterville Mine grid north is 13°20' west from true north and 21° west from magnetic north.

Table 2-2 Grid conversion reference points

Point 1: MIN5404 Mine Lease peg SE of Daley's Hill		
Coordinate System	**N**	**E**
GDA94 Zone 55	5930837.663	278011.932
Fosterville Mine Grid	4786.030	2177.630
Point 2: MIN5404 Mine Lease peg at NE corner		
Coordinate System	**N**	**E**
GDA94 Zone 55	5939047.136	278407.302
Fosterville Mine Grid	12713.150	4343.140

2.5 Mineral Resource and Reserve Definitions

The following definitions have been taken from the CIM definition standards for Mineral Resources and Reserves prepared by the CIM standing Committee on Reserve Definitions and Adopted by CIM Council on May 10, 2014.

CIM Definitions are underlined and defined terms referenced to NI 43-101 are double underlined.

2.5.1 Mineral Resources

Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource

A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth' s crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.

The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.
Material of economic interest refers to diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals.
The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of Modifying Factors. The phrase 'reasonable prospects for eventual economic extraction' implies a judgment by the Qualified Person(s) in respect of the technical and economic factors likely to influence the prospect of economic extraction. The Qualified Person(s) should consider and clearly state the basis for determining that the material has reasonable prospects for eventual economic extraction. Assumptions should include estimates of cutoff grade and geological continuity at the selected cut-off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing method and mining, processing and general and administrative costs. The Qualified Person(s) should state if the assessment is based on any direct evidence and testing.
Interpretation of the word 'eventual' in this context may vary depending on the commodity or mineral involved. For example, for some coal, iron, potash deposits and other bulk minerals or commodities, it may be reasonable to envisage 'eventual economic extraction' as covering time periods in excess of 50 years. However, for many gold deposits, application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time.

2.5.1.1Inferred Mineral Resources

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.
An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drill holes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Pre-Feasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101.
There may be circumstances, where appropriate sampling, testing, and other measurements are sufficient to demonstrate data integrity, geological and grade/quality continuity of a Measured or Indicated Mineral Resource, however, quality assurance and quality control, or other information may not meet all industry norms for the disclosure of an Indicated or Measured Mineral Resource. Under these circumstances, it may be reasonable for the Qualified Person to report an Inferred Mineral Resource if the Qualified Person(s) has taken steps to verify the information meets the requirements of an Inferred Mineral Resource.

2.5.1.2 Indicated Mineral Resource

An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person(s) when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person(s) must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Pre-Feasibility Study which can serve as the basis for major development decisions.

2.5.1.3 Measured Mineral Resource

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.

Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person(s) when the nature, quality, quantity and distribution of data are such that the tonnage and grade or quality of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability of the deposit. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

2.5.2 Modifying Factors

Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.

2.5.3 Mineral Reserves

A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility or Feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.

The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported.

The public disclosure of a Mineral Reserve must be demonstrated by a Pre-Feasibility Study or Feasibility Study.

Mineral Reserves are those parts of Mineral Resources which, after the application of all mining factors, result in an estimated tonnage and grade which, in the opinion of the Qualified Person(s) making the estimates, is the basis of an economically viable project after taking account of all relevant Modifying Factors. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the Mineral Reserves and delivered to the treatment plant or equivalent facility. The term 'Mineral Reserve' need not necessarily signify that extraction facilities are in place or operative or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

'Reference point' refers to the mining or process point at which the Qualified Person(s) prepares a Mineral Reserve. For example, most metal deposits disclose mineral reserves with a "mill feed" reference point. In these cases, reserves are reported as mined ore delivered to the plant and do not include reductions attributed to anticipated plant losses. In contrast, coal reserves have traditionally been reported as tonnes of "clean coal". In this coal example, reserves are reported as a "saleable product" reference point and include reductions for plant yield (recovery). The Qualified Person(s) must clearly state the 'reference point' used in the Mineral Reserve estimate.

2.5.3.1 Probable Mineral Reserve

A Probable Mineral Reserve is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource. The confidence in the Modifying Factors applying to a Probable Mineral Reserve is lower than that applying to a Proven Mineral Reserve. The Qualified Person(s) may elect, to convert Measured Mineral Resources to Probable Mineral Reserves if the confidence in the Modifying Factors is lower than that applied to a Proven Mineral Reserve. Probable Mineral Reserve estimates must be demonstrated to be economic, at the time of reporting, by at least a Pre-Feasibility Study.

2.5.3.2 Proven Mineral Reserve

A Proven Mineral Reserve is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve implies a high degree of confidence in the Modifying Factors.

Application of the Proven Mineral Reserve category implies that the Qualified Person(s) has the highest degree of confidence in the estimate with the consequent expectation in the minds of the readers of the report. The term should be restricted to that part of the deposit where production planning is taking place and for which any variation in the estimate would not significantly affect the potential economic viability of the deposit. Proven Mineral Reserve estimates must be demonstrated to be economic, at the time of reporting, by at least a Pre-Feasibility Study. Within the CIM Definition standards the term Proved Mineral Reserve is an equivalent term to a Proven Mineral Reserve.

2.5.4 Feasibility Study

A Feasibility Study is a comprehensive technical and economic study of the selected development option for a mineral project that includes appropriately detailed assessments of applicable Modifying Factors together with any other relevant operational factors and detailed financial analysis that are necessary to demonstrate, at the time of reporting, that extraction is reasonably justified (economically mineable). The results of the study may reasonably serve as the basis for a final decision by a proponent or financial institution to proceed with, or finance, the development of the project. The confidence level of the study will be higher than that of a Pre-Feasibility Study.

2.5.5 Preliminary Feasibility Study

A Pre-Feasibility Study is a comprehensive study of a range of options for the technical and economic viability of a mineral project that has advanced to a stage where a preferred mining method, in the case of underground mining, or the pit configuration, in the case of an open pit, is established and an effective method of mineral processing is determined. It includes a financial analysis based on reasonable assumptions on the Modifying Factors and the evaluation of any other relevant factors which are sufficient for a Qualified Person, acting reasonably, to determine if all or part of the Mineral Resource may be converted to a Mineral Reserve at the time of reporting. A Pre-Feasibility Study is at a lower confidence level than a Feasibility Study.

2.5.6 Mineral Resource And Mineral Reserve Classification

The CIM Definition Standards provide for a direct relationship between Indicated Mineral Resources and Probable Mineral Reserves and between Measured Mineral Resources and Proven Mineral Reserves. In other words, the level of geoscientific confidence for Probable Mineral Reserves is the same as that required for the in situ determination of Indicated Mineral Resources and for Proven Mineral Reserves is the same as that required for the in situ determination of Measured Mineral Resources. , Figure 2-1 displays the relationship between the Mineral Resource and Mineral Reserve categories.

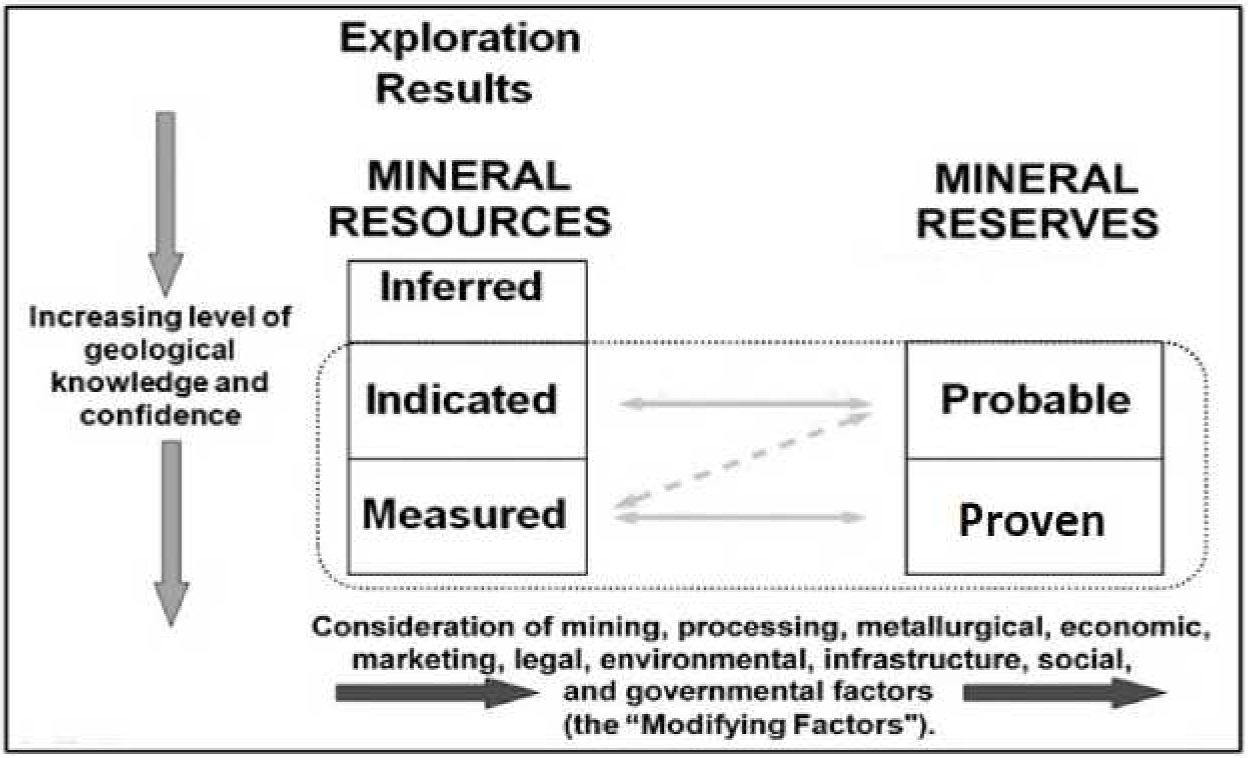


Figure 2-1 Conversion Of Mineral Resources To Mineral Reserves (CIM Standards 2014)

3 RELIANCE ON OTHER EXPERTS

The Qualified Persons have prepared this report from a range of sources including their personal work, contributions, from other FGM personnel and reports from a range of external consultants. Where input has been received from these sources, the Qualified Persons have reviewed and verified the contained assumptions and conclusions. The Qualified Persons do not disclaim responsibility for this information.

3.1 Contributing Authors

Troy Fuller (Geology Manager ‐ Fosterville Gold Mine) BSc (Geology) Hons, MAIG is the site based Qualified Person for the Central and Harrier Areas. He has contributed to information contained in Sections 1-14, 17, 18.2 -27 and 28.2 of this report.

Ion Hann (Mining Manager ‐ Fosterville Gold Mine) BEng (Mining), FAusIMM is a site based Qualified Person and has made contributions to Sections 15-16, 18.1 and 28.1 of this report.

4 PROPERTY, DESCRIPTION AND LOCATION

The FGM is located about 20km north east of Bendigo and 130km north of Melbourne in the State of Victoria, Australia (Figure 4-1).

Figure 4-1 Fosterville Project Location Map.

The FGM and all associated infrastructure including the tailings dam and waste dumps are located on Mining Lease 5404 (MIN5405; Figure 4-2), which is 100% owned by Newmarket Gold Inc. MIN5404 was initially granted as ML1868 on 24th August 1990. The licence later merged with adjoining lease MIN4877, resulting in MIN5404.

In December 2012 another mining lease (MIN5565) was granted to FGM, and this licence was also merged into MIN5404. The present MIN5404 has a total area 17.157 km², and is active until August 24, 2020.

MIN5404 is located at centroid coordinates 276,599.72mE and 5935,134.9mN using Map Grid of Australia Zone 55 (GDA94) coordinate projection (or 144° 29' 56.9" Longitude and -36° 42' 11.6" Latitude).

The boundaries of land covered by the Mining Licence are accurately surveyed and marked on the ground with posts, trenches and information plates in accordance with the Mineral Resources Development Regulations 2002.

Newmarket Gold also holds titles through FGM of two surrounding exploration licences totaling 504.9km ^{2}. These exploration licences encompass the entire known strike extent of the Fosterville Goldfield. In the State of Victoria, exploration licences are renewable annually subject to adequate exploration expenditure and statutory licence size reductions. However, EL3539 is recognized by the State of Victoria as a strategic licence and is able to be renewed on an annual basis without size reductions until 26th February, 2017 (Figure 4-2). To retain the two tenements Newmarket Gold is required to conduct exploration programs and commit to minimum annual expenditures that are prescribed by Earth Resources Regulation Victoria. Presently, the annual expenditures are set at $188,700 for EL3539 and $34,000 for EL4937.

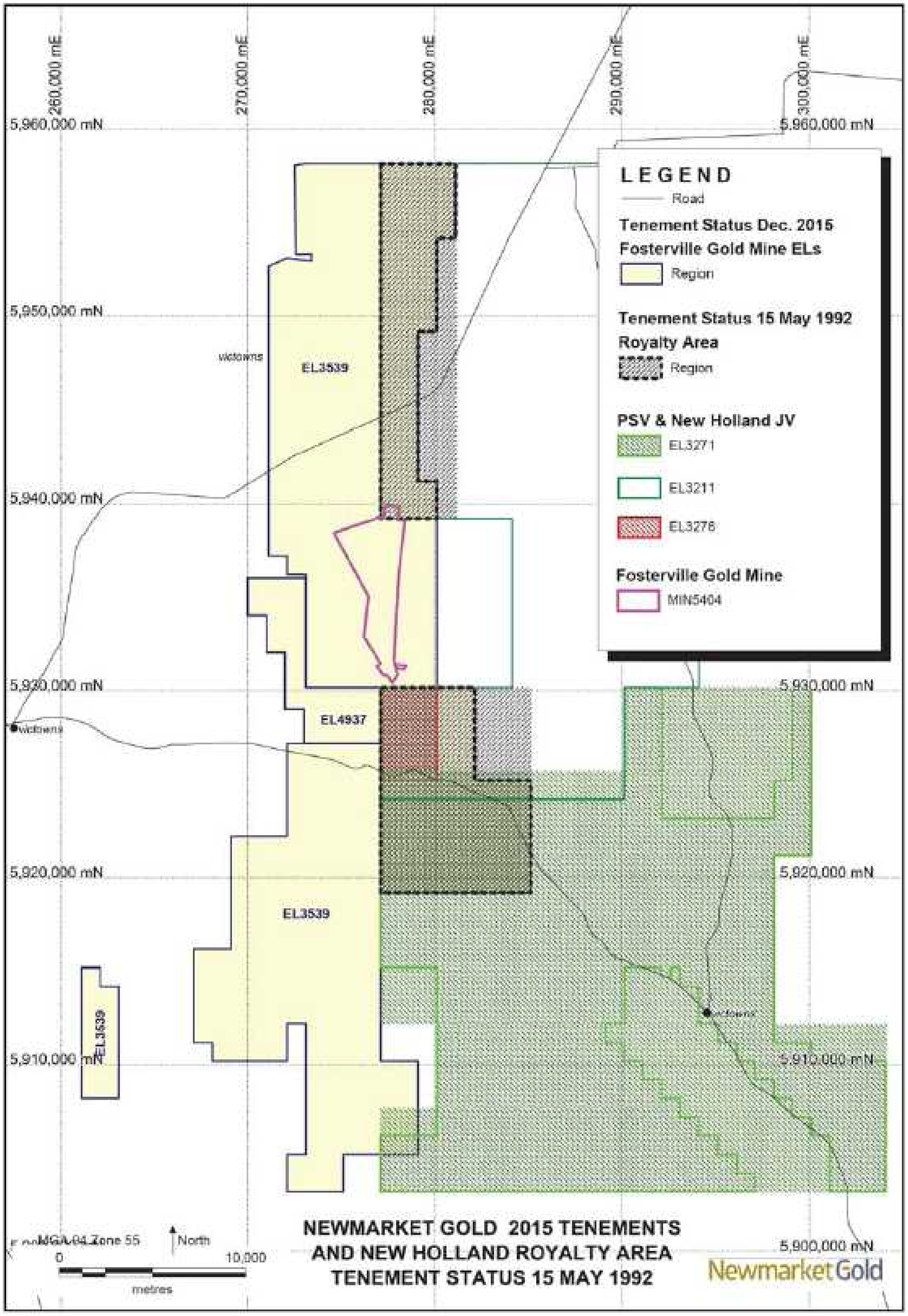


Figure 4-2 Fosterville mining lease plan with exploration leases and royalty areas.

Within MIN5404, there is a 2.5% gold royalty payable to New Holland Mining Ltd, now Nu Energy Capital Limited for the area outlined by an historical mining lease MIN4877 in the north eastern portion of MIN5404. Furthermore, the royalty agreement extends north and south of MIN5404 where previously existing tenements EL3211, EL3271 and EL3276 (New Holland Mining) overlap with EL3539 (FGM). See Figure 4-2.

When Crocodile Gold acquired the Fosterville and Stawell Gold Mines from AuRico in 2012, a net free cash flow sharing arrangement was established where Crocodile Gold was entitled to cumulative net free cash flow from those mines of up to C$60 million. AuRico would then be entitled to 100% of the next C$30 million in net free cash flow, after which Crocodile Gold and AuRico would share the next C$30 million of net free cash flow on a 50/50 basis until C$120 million of cumulative net free cash flow was achieved, following which AuRico would be entitled to 20% on an ongoing basis.

On December 22, 2014 it was announced that Crocodile Gold had reached a mutually beneficial agreement with AuRico that terminated their net free cash flow sharing arrangement in exchange for a one-time payment of C$20 million in cash and a net smelter return royalty of 2% from Fosterville Gold Mine (effective upon final approval from the Foreign Investment Review Board of Australia) and a 1% royalty from the Stawell Gold Mines (commencing January 1st 2016), releasing Crocodile Gold from its obligation to pay AuRico any further net free cash flow generated from its Victorian operations. This agreement means that Newmarket Gold is obligated to pay AuRico a net smelter royalty of 2% from Fosterville Gold Mine. However, Alamos Gold Inc. (Alamos) merged with AuRico in July 2015, which has resulted in Newmark Gold now being obliged to pay the new company, AuRico Metals, the net smelter royalty of 2% from Fosterville Gold Mine.

There are no state government royalties on gold production in the State of Victoria.

There are no native title issues relevant to MIN5404.

The environmental bond is currently set at AUD$7.84M and is reviewed annually with the Department of Economic Development, Jobs, Transport and Resources Victoria. Rehabilitation is undertaken progressively at the FGM but the environmental bond is only reduced on establishment of the rehabilitation, which is not considered to have occurred until at least 5 years after rehabilitation has been undertaken.

The FGM is located near areas of moderate environmental significance (Mt Sugarloaf Nature Conservation Reserve), established productive farmland and is adjacent to the locally significant Campaspe River.

The FGM is operating under a Work Plan approved in April 2004 under Section 44(1) of the Mineral Resources Development Act. The approval, concerning MIN5404 (formerly ML1868), MIN4456 and MIN4887, was given by the Minister of Environment and Water at that time to Perseverance Exploration Pty Ltd. Work Plan Variations are submitted where significant changes from the Work Plan exist.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The Fosterville area is flat to very gently undulating with a range of low, rolling hills about 2km to the west and the Campaspe River about 2km to the east. On MIN5404, natural surface elevations range from 150m to 185m above sea level (5150mRL to 5185mRL mine grid). Vegetation in the area ranges from native forest to established grazing pasture.

The FGM has ready access via two separate sealed roads and a variety of all-weather un-sealed roads linking to regional highways. The regional center of Bendigo is approximately 20km away and has a population of around 113,000, which provides a source of skilled labour. The area has a Mediterranean climate with hot, dry summers and cool winters. The average climate statistics from the nearby Bendigo Airport weather station for the period 1991-2015 are listed in Table 5-1 below.

Table 5-1 Climate statistics for Bendigo Airport, 1991-2015 (Bureau of Meteorology, 2016).

Statistics	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Temperature												
Mean maximum temperature (°C)	29.8	29.6	25.8	21.2	16.6	13.4	12.5	14.2	17.0	20.8	24.6	27.4
Mean minimum temperature (°C)	4.1	14.4	11.5	7.8	5.2	3.6	2.6	2.8	4.5	6.6	9.7	11.8
Rainfall												
Mean rainfall (mm)	34.6	33.9	29.4	30.5	44.9	52.6	54.2	50.1	49.9	39.2	46.1	40.0
Decile 5 (median) rainfall (mm)	22.9	22.8	18.2	23.6	33.3	47.6	57.8	48.2	45.4	32.0	42.6	30.4
Mean number of days of rain ≥ 1 mm	3.8	3.3	3.5	4.1	6.3	7.6	9.1	8.2	7.1	5.4	5.8	4.6

Power is supplied to the site via a terminal station that was constructed by PSV in 2005. This station is connected to the 220kV transmission line that runs from Bendigo to Shepparton and traverses the southern end of MIN5404 approximately 2 km south of the processing plant. There is a connection agreement in place with SP Ausnet who manages the transmission and distribution network. To improve the security of water supply, an agreement was reached for the supply of waste water from the Bendigo sewerage treatment facility. A pipeline was commissioned in April 2005 that has the capacity to supply approximately 2,000ML annually, which comfortably exceeds the current plant usage of approximately 1,000ML per annum. The agreement was for an initial ten year term with two options of a further ten years each on written request.

All other site infrastructure is in place and approved in the Work Plan established in April 2004.

Details of tailings storage areas are covered in sections 18.1.4 and 20.2 and the location of tails storage facilities is illustrated in Figure 18-1, Figure 18-3 and Figure 18-4.

The location and layout of the processing plant site is illustrated in Figure 18-1 and Figure 18-2. The layout of the comminution circuit allows for installation of a pebble crushing circuit should it be required, and a secondary ball mill to increase grinding circuit capacity. Space was left in the area layouts for additional tank farms and equipment to accommodate a nominal increase in plant capacity. Space exists to the east of the plant site to duplicate existing facilities to double plant throughput if required.

Mining waste material that cannot be placed underground is brought to the surface and dumped within the confines of the Ellesmere pit (Figure 18-1) (Section 18.2.4) . Details on the storage of historically mined waste overburden is covered in section 20.2 and tabulated in Table 20-1

6 HISTORY

Gold was first discovered in the Fosterville area in 1894 with mining activity continuing until 1903 for a total of 28,000 ounces of production. Mining in this era was confined to near-surface oxide material.

Aside from a minor tailings retreatment in the 1930' s, the field lay dormant until the 1988 when Bendigo Gold Associates recommenced gold production at Fosterville from the reprocessing of tailings. By 1989 this program had come to an end and exploration for oxide resources commenced. The leases were then acquired by Brunswick who continued exploration and in 1991 started heap leaching ore derived from shallow oxide open pits. After six months of production, Brunswick went into receivership as a result of the failure of another operation. Perseverance bought the operation from the receivers and continued the oxide heap leach operations. Perseverance continued to produce between 25,000 ounces to 35,000 ounces per annum until the cessation of the oxide mining in 2001. Between 1988 and 2001, a total of 240,000 ounces of gold were poured (Roberts *et al*, 2003).

In 2001, Perseverance underwent a significant recapitalization and the focus of the company changed to developing the sulphide resource. A feasibility study investigating a combined open pit and underground mining operation feeding 0.8Mtpa of sulphide ore to a BIOX® processing plant was completed in 2003. Work on the plant and open pit mining commenced in early 2004. Commercial sulphide hosted gold production commenced in April 2005 and up to the end of December 2006 had produced 136,882 ounces of gold. Underground development commenced in March 2006 with first production recorded in September 2006 and significant open pit production ceasing at the end of 2007, but with minor production from open pits in 2011 and 2012. The 500,000th ounce milestone of 'sulphide' gold production was achieved in April 2011 and by the end of December 2015 'sulphide' gold production totaled 1,000,682 ounces.

A breakdown of open cut and underground mined tonnes and grade since the commencement of the sulphide operation is given in Table 6-1.

Table 6-1 Mined production data for Fosterville for the period 2004-2015.

Mining Area		2004	2005	2006	2007	2008	2009	2010	2011	2012	2013	2014	2015
Open Cut	Tonnes (kt)	52	517	1,084	423	-	-	-	45	75	-	-	-
	Grade (g/t Au)	3.6	5.6	3.4	2.3	-	-	-	2.8	2.6	-	-	-
Under-ground	Tonnes (kt)	-	-	36	376	512	780	729	734	729	827	786	704
	Grade (g/t Au)	-	-	4.8	4.2	4.5	4.8	5.0	5.0	4.5	4.6	4.6	6.1
Total	Tonnes (kt)	52	517	1,120	799	512	780	729	779	804	827	786	704
	Grade (g/t Au)	3.6	5.6	3.4	3.2	4.5	4.8	5.0	4.9	4.3	4.6	4.6	6.1

On 29th October 2007, Perseverance announced that it had entered into an agreement with Northgate Minerals Corporation (Northgate) to acquire the company via a Scheme of Arrangement. This agreement was ratified by Perseverance' s shareholders and option holders on the 18th January 2008 with full control passing to Northgate in February 2008.

In August 2011 Northgate entered into a merger agreement with AuRico, who assumed control of the Northgate assets in October 2011. However, in March 2012 AuRico and Crocodile Gold jointly announced that Crocodile Gold would acquire the Fosterville and Stawell mines. Crocodile Gold' s ownership of Fosterville was achieved on May 4, 2012. In May 2015 Crocodile Gold and Newmarket Gold Inc. entered into a definitive arrangement agreement and completed a merger on the 10th of July 2015 to form Newmarket Gold.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The western sub-province of the Paleozoic Lachlan Orogen in Victoria has been divided into three major fault-bounded structural zones: the Stawell, Bendigo, and Melbourne Zones (Figure 7-1a; Caley et al, 2011). These structural zones are dominated by chevron-folded Cambro-Ordovician to Devonian turbidite sequences, and were progressively intruded by Early Silurian granite plutons in the west, through to Late Devonian granite plutons in the East (Bierlein & McKnight, 2005; Phillips et al. 2012).

The Fosterville Goldfield is located within the Bendigo Zone, which is bounded by the Avoca Fault to the west and the Heathcote Fault Zone to the east (Figure 7-1b), both of which are steep west-dipping reverse faults. The Bendigo Zone contains thick Ordovician turbidite sequences that were subjected to low grade metamorphism during Late Ordovician Benambran Orogeny (~455-440 Ma) and the Late Devonian Tabberabberan Orogeny (~380 Ma). East-vergent folding and thrusting indicates a predominantly east-west compression that resulted in the formation of north-south upright folds. Continued deformation caused steepening of fold limbs and progressive development of a series of west-dipping reverse faults. These faults are interpreted to have listric geometries at depth and were likely conduits that provided a regional control on mineralizing processes, in conjunction with intrazonal west dipping faults, such as the Redesdale Fault mapped south of Fosterville (Cayley et al, 2008). In addition, smaller reverse faults propagated across fold limbs, linking bedded faults and are well mineralized in the style characteristic to the classic Central Victorian Slate Belt Gold Deposits of Bendigo and Castlemaine (Roberts et al, 2003).

Gold mineralization is associated with to two main events across the western Lachlan Orogen at ~445Ma and ~380-370Ma, with a possibly another minor event at ~410-400Ma (Phillips et al., 2012). The ~445Ma event is thought to have involved crustal thickening and the circulation of metamorphic fluids through the crust (Vandenberg et al, 2000) and formed gold deposits at Bendigo, Castlemaine, Maldon and Daylesford. The ~380-370Ma event is restricted largely to the Melbourne and eastern-Bendigo Zones and is responsible for the emplacement of gold at the Fosterville Goldfield (Bierlein and Maher 2001). The minor period of mineralization at ~410-400 Ma is restricted to the Stawell and western Bendigo Zones and associated with crustal anatexis and Early Devonian plutonism (Phillips et al., 2012). The two major gold mineralizing events have been linked to the Benambran and Tabberabberan Orogenies (VandenBerg et al., 2000). All three gold mineralizing events are characterized by carbonate and sericite alteration, but only the latter two events (~410-400Ma & ~380-370Ma) have elevated Mo, Cu, Sb and W. During the third mineralizing event a range of mineralization styles resulted and include quartz-carbonate vein hosted free gold through to sulphide hosted refractory gold in association with arsenopyrite, pyrite and stibnite (Roberts et al, 2003).

Deep weathering and erosion in the late Tertiary resulted in the development of a regional laterite profile with weathering locally to 50m depth.

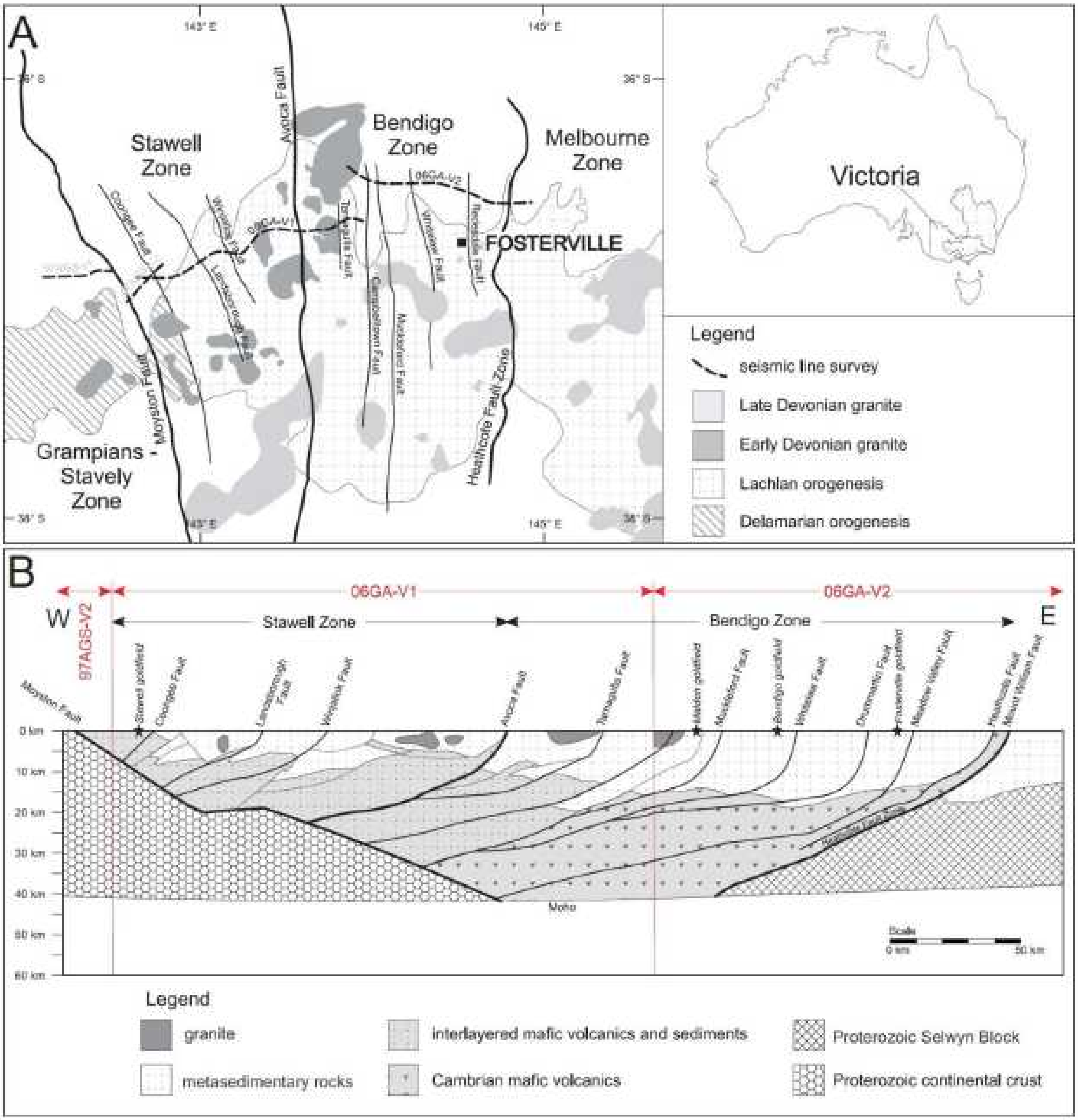


Figure 7-1 Map and cross section of the Western Lachlan Fold Belt in central Victoria.
(a) Distribution of major geologic units and major faults of the Bendigo and Stawell Zones and location of seismic lines.
(b) Geological interpretation from seismic surveys. Adapted from Leader & Wilson, 2010.

7.2 Local and Property Geology

The Fosterville Goldfield is hosted by Lower Ordovician Lancefieldian (486~488 Ma) turbidites within the Ordovician Castlemaine Group rocks (Figure 7-2 and Figure 7-3) The turbiditic sequence comprises interbedded sandstones, siltstones and shales, which are interpreted as having formed in a meandering submarine channel setting. The sequence is dominated by shale topped sands ranging from 0.2m to 1.5m in thickness, with lesser amounts of massive sandstone, shale and black shale (Roberts et al, 2003). Detailed drill core logging has confirmed almost 1km of stratigraphic succession exists at Fosterville and correlation of sedimentary units has been possible over a 10km distance within the Fosterville Mine lease (Boucher et al, 2008a).

The sequence is metamorphosed to sub-greenschist facies, or more precisely, Anchizone to lower Epizone (Melling, 2008) and fluid inclusion work indicates that the Fosterville Goldfield formed at ~270°C and at 2.6 -5.7km crustal levels (Mernagh, 2001).
The stratigraphic sequence was folded into a set of upright chevron, occasional open style folds, with fold wavelengths in the order of 350m and parasitic fold wavelengths of 50m. During folding vertical axial planar (in finer sediments) and radial cleavages (sandstones) developed and are best observed in fold hinges. Bedded LQ veins (<1mm to 0.8m wide) were also formed during early folding and were preferentially formed in shales and at or close to the contact with sandstone units.
The north-south trending Redesdale Fault, lying approximately 2km to the east of the Fosterville area, is an important intrazonal fault and occurs in the hangingwall of the Heathcote Fault Zone (Figure 7-1a and Figure 7-2)
Subordinate faults (third or fourth order), such as the Fosterville, O'Dwyer's and Sugarloaf Faults (Figure 7-2) all have associated gold mineralization and are located in the hangingwall of the Redesdale Fault.
Within the Fosterville area the north-north-west trending Fosterville Fault is strike extensive and dips steeply west. In its footwall are moderately west dipping faults with varying reverse offsets combing with reactionary east dipping faulting to create zones of fractured wallrock and sulphide mineralization. In general faults with greater reverse offset have larger gold mineralization dip and plunge lengths; and where faulting is more complex, wallrock fracturing is enhanced and mineralization width increases.
A fold culmination (dome) exists in the Fosterville Mine Lease in Falcon pit area (Figure 7-3). about which a fold plunge reversal occurs. South of the culmination, folds plunge approximately 20° southwards, and a large west-dipping fold limb, containing parasitic folds and faulting has been well drilled over a 4km length to as far south as Daley's Hill. Extensive drilling focused on south plunging gold mineralization associated with late brittle west dipping reverse faulting that offsets syncline and anticline fold closures (Figure 7-5)
In the northern portion of the Mine Lease, in the Robbin's Hill - O'Dwyer's area, a number of west dipping faults occur and parallel the Fosterville Fault. Late Silurian to Early-Devonian porphyry dykes (Arne et al, 1998) also occur in this area, are up to 10m in width, intrude the stratigraphic sequence along anticlinal axial planes (King, 2005 & Reed, 2007a) and postdate all significant faulting, The porphyry dykes are sericite altered, have associated gold mineralization that was sufficient to support several oxide and minor sulphide (O' Dwyer's South) open pits.
Lamprophyre dykes, typically less than 1m in width, intrude along the general Fosterville Fault trend and are unmineralized. These dykes were emplaced in the Middle Jurassic (157-153 Ma) (Bierlein et al, 2001) and are of similar age to those that occur at Bendigo.

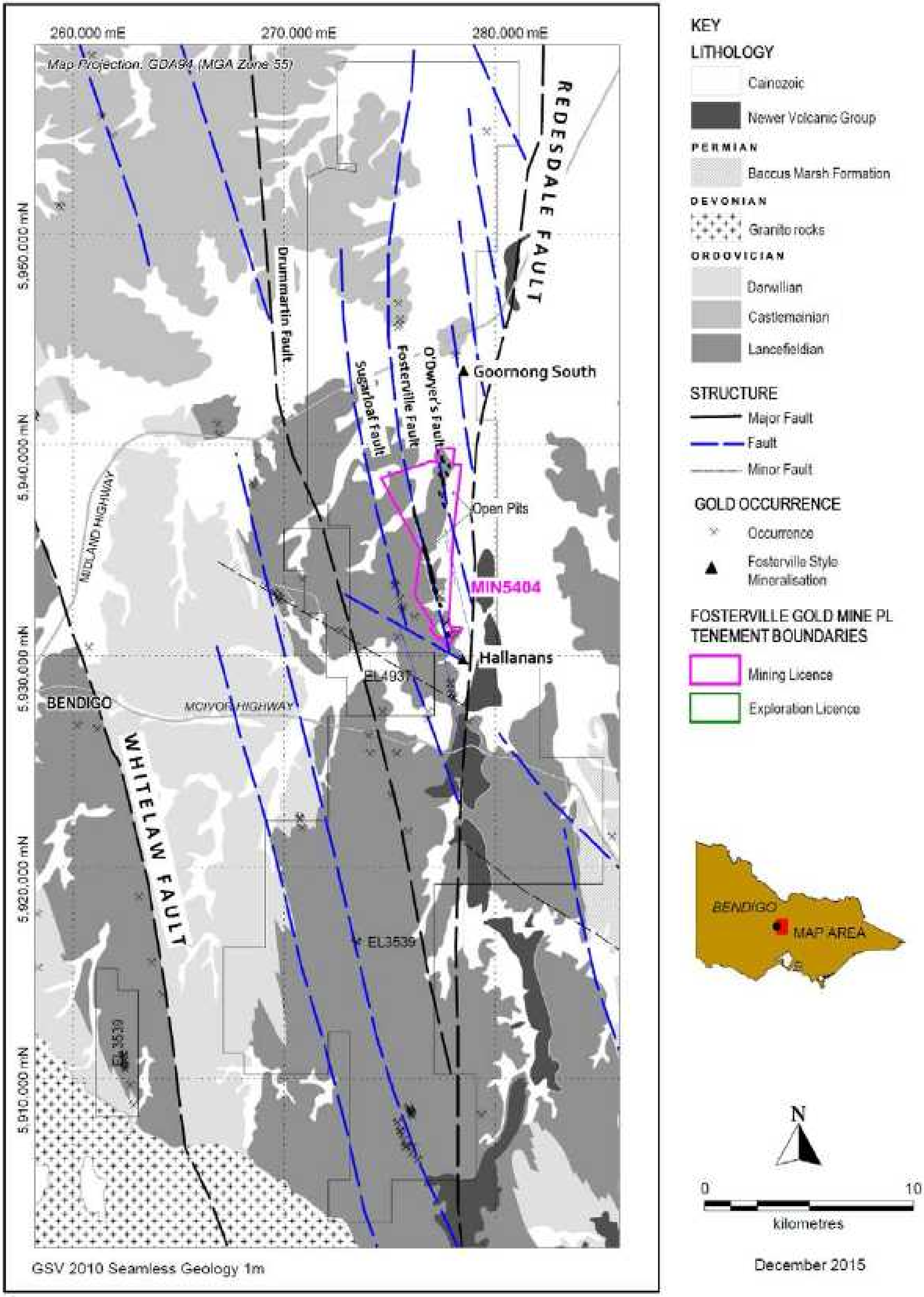


Figure 7-2 Regional Geology Plan of the Fosterville District.
Fosterville Mining Licences, Exploration Licences, open pits and hard rock gold occurrences

Erosion of the area followed by Cainozoic Murray Basin sediment valley backfill and weathering has resulted in local clay conglomerate alluvial channels and complete oxidation to about 40m below surface. Immediately below the base of complete oxidation is a 10m to 15m thick zone of partial oxidation of sulphide minerals. Feldspar destruction and partial carbonate dissolution extends from the base of oxidation to about 150m depth. Approximately 2km to the east of Fosterville Miocene aged Newer Basalt Group rocks mask the Ordovician rocks and Murray Basin sediments.

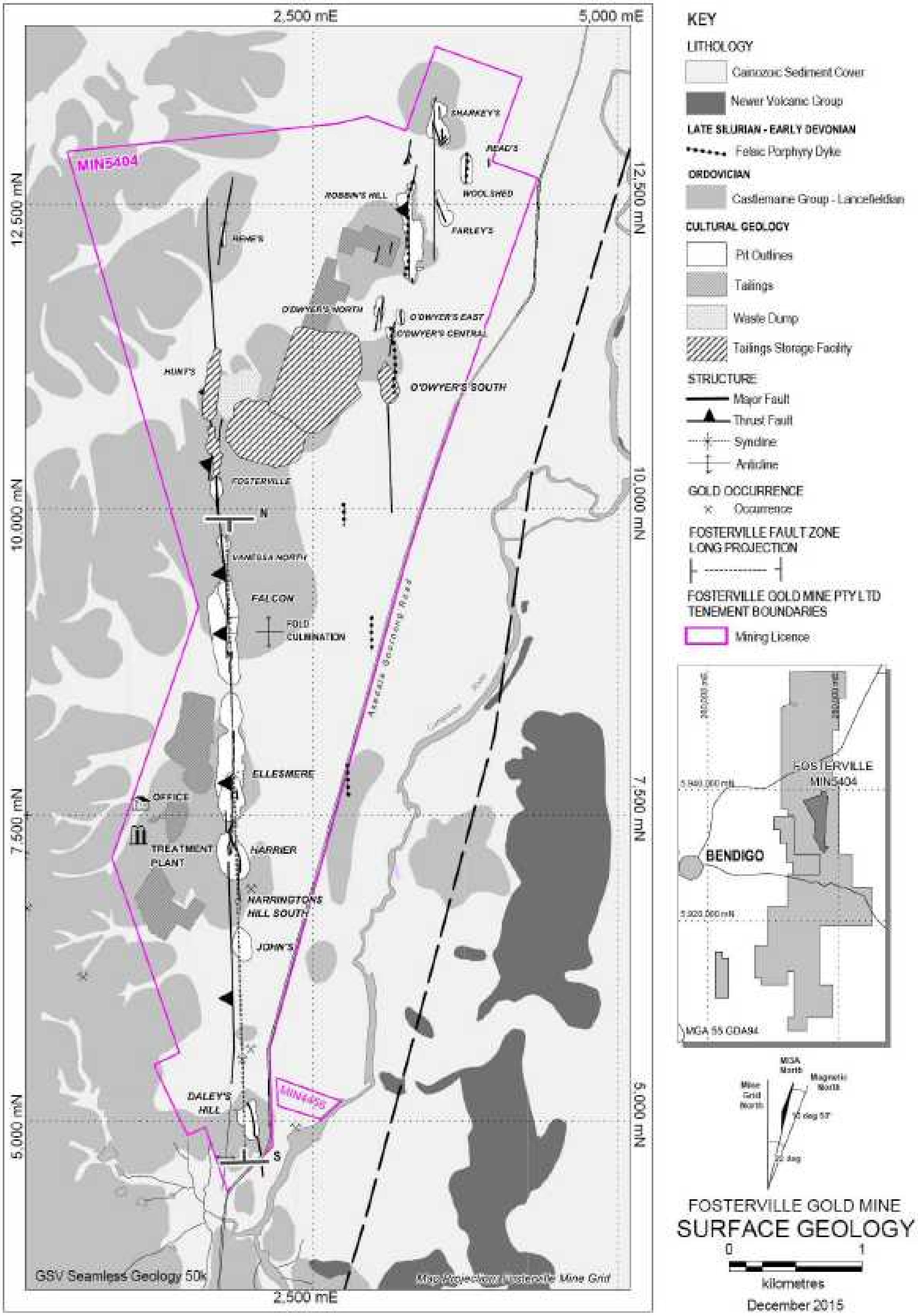


Figure 7-3 Fosterville Surface Geology Plan showing surface mining activity.

Schematic Geological Cross Section

The geological knowledge of the Fosterville Fault Zone fault architecture has progressively grown over the last decade as diamond drilling explored new areas and underground mining reached deeper levels. The present understanding of the faulting is shown on schematic cross sections (Figure 7-4 and Figure 7-5). Pictured is the moderate-steep west dipping Fosterville Fault, which has several en echelon array of footwall reverse faults that link across from a western anticline to a syncline in the east.

The en echelon array of reverse faults has a general form and is described from west to east.

Most of the lower faults (Hawk through to Kestrel) are thought to exist as bedded LQ veins at depth to the west of their respective footwall anticlines. However, eastwards between footwall and hangingwall anticlines the faults have concordant (parallel)/discordant (oblique) bedding relationships and to the east of hangingwall anticlines, the faults shallow in dip and have discordant contacts with adjacent bedding. When certain stratigraphic units are encountered across the east dipping limb, reactionary east dipping structures form, creating zones of greater structural complexity. Further eastwards the single stranded west dipping faults become an unmineralized zone of distributed faults for 50-100m, before merging into a single fault, approximately 50m west of footwall synclines. East of the footwall syncline the faults' dip steepens, matching the dip of the footwall bedding. Between footwall and hangingwall synclines, faults have discordant/ concordant bedding relationships and to the east of the hangingwall syncline the faults exist as bedded LQ veins, commonly with pug on one margin.

Structurally higher level faults such as the Harrier and Osprey Faults appear as footwall faults emanating/ splaying from the footwall of the Fosterville Fault.

The schematic cross section portrays a number of fault segments where gold mineralization occurs and includes examples of areas of fault-bedding discordant relationships, changes in fault dip and localization of mineralization between hangingwall and footwall synclines and to a lesser extent between hangingwall and footwall anticlines. In particular, the Phoenix Fault System is the most important structure at Fosterville for gold mineralization, has 120-150m of reverse offset and as underground mining has progressed to deeper levels, faulting becomes more complex. Nearer to surface the Phoenix Fault was a relatively simple single stranded west-dipping reverse fault. However, down-plunge the faulting changes to also include mineralized hangingwall splay faulting and west dipping footwall faults emanating from bedding parallel LQ veins.

Other faults at structurally higher positions have comparable fault offset and are well mineralized. These include the Harrier and Osprey Faults (exposed at Harrier Pit) that are footwall splays of the Fosterville Fault. The faults have over 200m of combined reverse movement, and are mined at the southern end of the mine lease.

Where wall rocks are faulted and brecciated, fractures are healed by quartz-carbonate veining and commonly have arsenopyrite and pyrite disseminated in the wall rock up to 50cm from the veins. The wall rock proximal to faults is also sericitised, sometimes with alteration visually subtle, and has similar spatial extents to the gross disseminated sulphide distribution. Bedded faults exist as LQ veins and are generally poorly mineralized.

Drilling at Fosterville has generally been concentrated in a 200m zone east of the Fosterville Fault to where footwall faults become bedded.

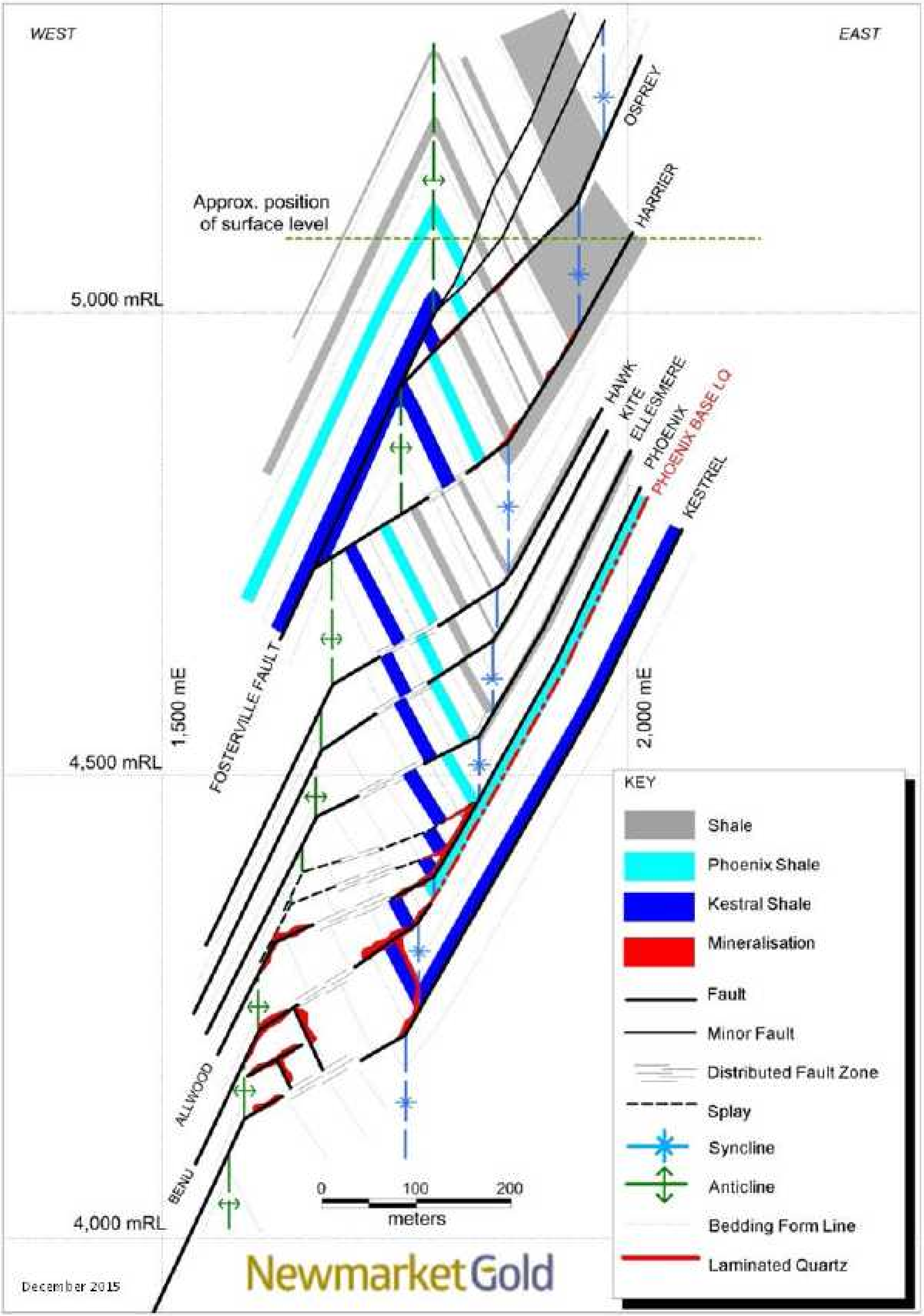

WEST
EAST
OSPREY
HARRIER
Approx. position of surface level
5,000 mRL
HAWK
KITE
ELLESMERE
PHOENIX
PHOENIX BASE LQ
KESTREL
1,500 mE
2,000 mE
FOSTERVILLE FAULT
4,500 mRL
ALLWOOD
BENU
0
100
200
meters
4,000 mRL
December 2015
Newmarket Gold
KEY
Shale
Phoenix Shale
Kestral Shale
Mineralisation
Fault
Minor Fault
Distributed Fault Zone
Splay
Syncline
Anticline
Bedding Form Line
Laminated Quartz

Figure 7-4 Fosterville Fault Zone schematic cross Section.

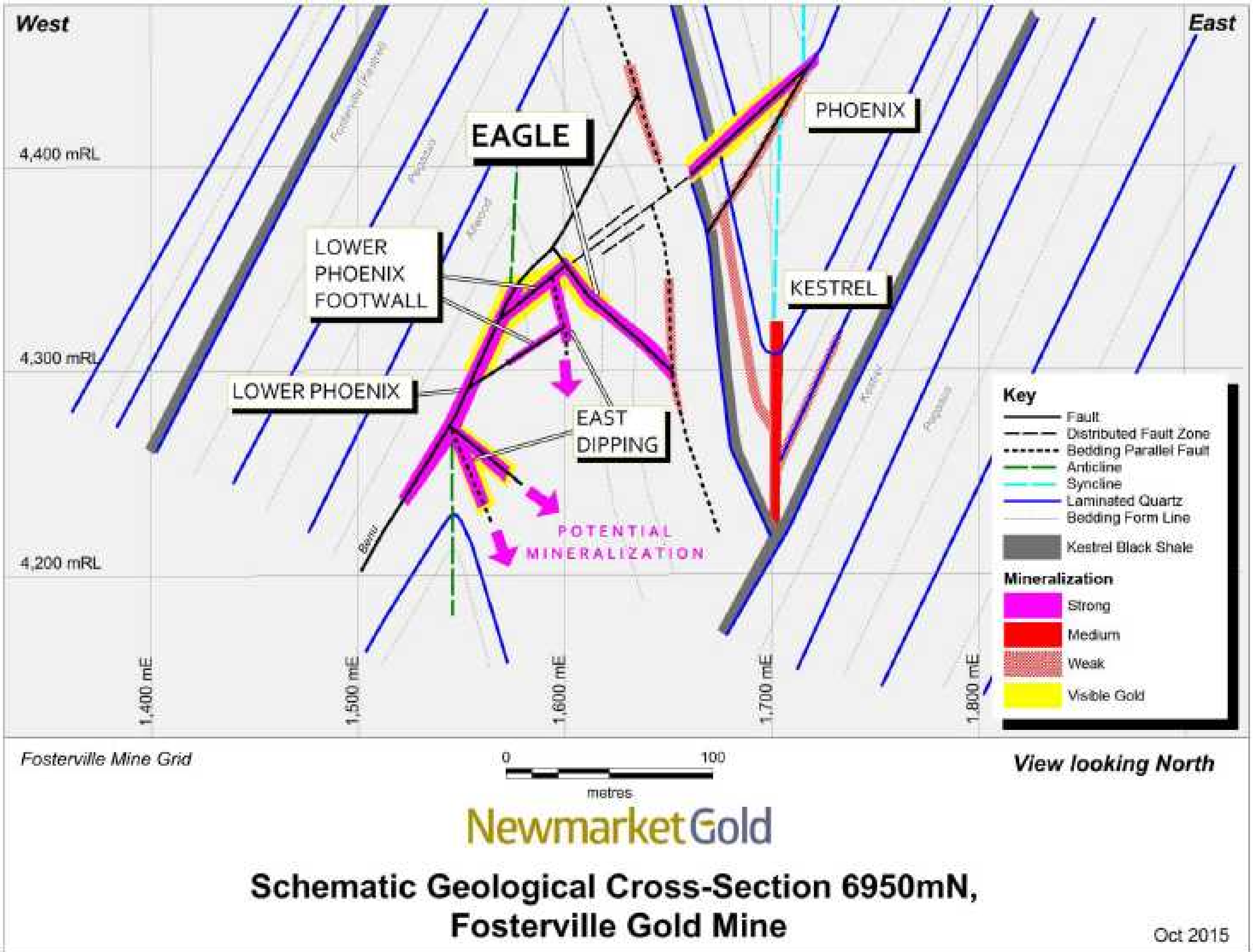


Figure 7-5 Schematic geological cross-section of 6950mN.

7.3 Mineralization

Primary gold mineralization at Fosterville occurs mainly as gold within disseminated arsenopyrite and pyrite forming as a selvage to veins in quartz‑carbonate veinlet stockwork, which is in turn controlled by the late brittle faults. Lesser amounts of primary gold, occurs as an overprint in the form of visible gold in quartz and has a spatial association with stibnite mineralization

The arsenopyrite occurs as 0.05 -6mm long acicular needles with no preferred orientation. The disseminated pyrite associated with gold mineralization occurs as crystalline pyritohedrons 0.1 -2mm in size. Electron microprobe analyses and metallurgical test work indicates that the arsenopyrite contains 100-1000 g/t Au and the auriferous pyrite 10-100 g/t Au (Roberts et al, 2003). Approximately 80% of the gold occurs in arsenopyrite, with the remaining 20% hosted by pyrite.

The quartz‑carbonate veinlet stockwork comprises a network of tension gash type quartz‑carbonate veinlets, which have formed perpendicular to the walls of the brittle faults and quartz‑carbonate veinlets formed on minor slip planes parallel to the brittle faults. Further movement on the minor slip planes offsets the tension gash veinlets giving rise to a range of geometries from planar through to highly erratic. The quartz‑carbonate veinlets are barren but have selvages of disseminated, fine grained arsenopyrite ‑ pyrite. Where the stockwork is well developed, mineralization selvages merge forming a solid body of mineralization. On the margins of the stockwork the mineralization occurs as a discrete selvage about 10 times the width of the veinlet on which it is centered.

Antimony mineralization, in the form of stibnite, occurs with quartz and varies from replacement and infill (up to several metres in width) of earlier quartz-carbonate stockwork veins, to massive stibnite-only veins of up to 0.5m in width. The stibnite-quartz zones are commonly associated with the Phoenix, Lower Phoenix, Lower Phoenix Footwall and Eagle with infrequent exposure within Harrier and the O'Dwyer's and Harrier Open Pits. Figure 7-6 illustrates antimony mineralization within an east dipping quartz-carbonate vein.

Primary visible gold is found within the Phoenix, Lower Phoenix and Eagle Mineralized Zones and there is a noticeable increase of visible gold occurrences at depth and down plunge in these structures (Figure 7-5). Gold nuggets, ≤3mm in size, are observed in drill core and underground development mapping, where specks of gold are define narrow linear trends. Visible gold has also been observed to form as isolated specks without a clear trend and are associated with vuggy quartz carbonate veins up to several meters in width (Figure 7-7). Minor primary visible gold has also been recorded in the O'Dwyer's South areas and at depth below Daley's Hill

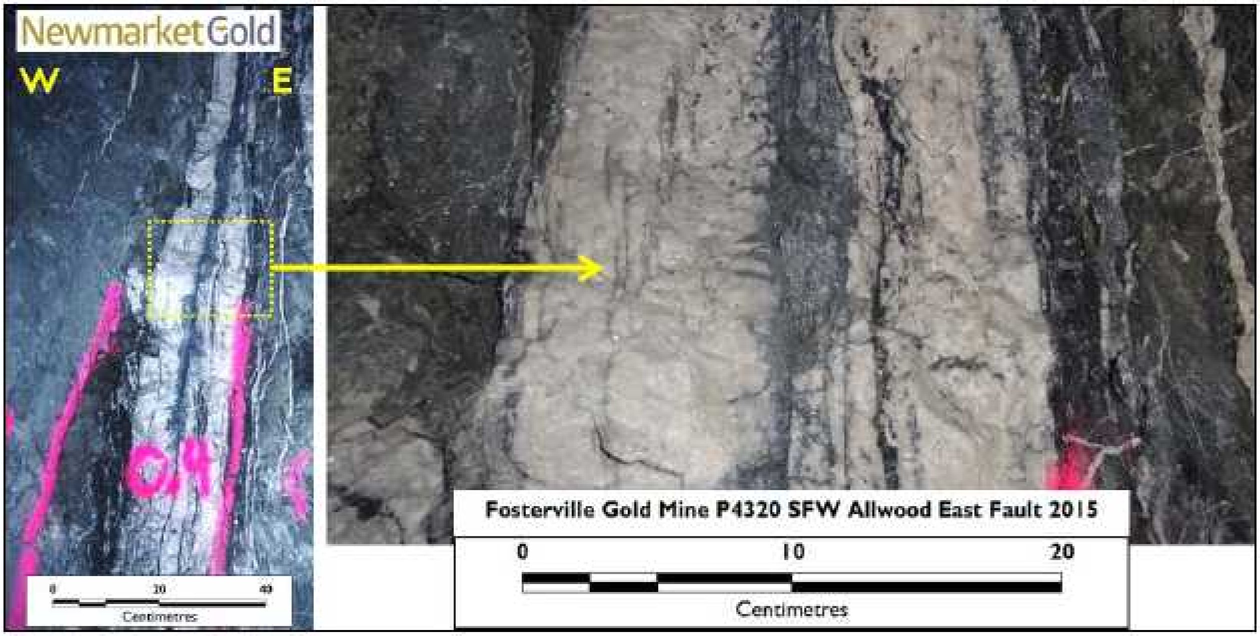


Figure 7-6 Underground face photo of the P4320 South Footwall Development showing stibnite overgrowth of quartz carbonate veining on the Allwood East Fault.

Figure 7-7: Drill core from hole UDH1221 showing relationship between visible gold (red circles) and vugs (blue circles)

The visible gold has a spatial association with stibnite in fault-related quartz veins; however, the stibnite mineralization does occur without visible gold (Henderson 2014). The rationale for the one-way correlation is likely due to the mineralization occurring in different events, but utilizing the same structurally favorable locations.

Several internal studies have been conducted to better understand different aspects in regards to the nature of visible gold mineralization. These studies include but are not limited to:

Wall sampling: ICP analyses were conducted on samples taken from an access drive underground (Phoenix 4340) in a zone that was observed to be strongly mineralized with stibnite and visible gold. Desktop studies were then undertaken to test for correlations between visible gold grade and chemical elements that were analyzed for exists. No such correlations were found.

Scanning Electron Microscope (SEM): A hand sample of wall rock was analyzed from an access drive underground (Phoenix 4340) using SEM at Monash University. The quartz veining was unusual in that it had a slightly pink coloration to it. SEM analysis concluded it was due to the presence of ankerite and dolomite.

Quartz comparison: A study was conducted to assess whether any other element other than gold could be used as a proxy to visible gold, and help define visible gold bearing mineralization . Elements included Pt, Al, As, Ag, Ba, Be, Bi, Ca, Co, Cr, Cu, Fe, K, Mg, Na, Pb, S, Sb, Sn, Ta, Ti, Tl and Zr were analyzed using combinations of fire assay, ICP and aqua regia digest. This study compared quartz that was proximal to visible gold against quartz that was inferred to not be related to visible gold. Findings were inconclusive.

Fire assay to completion: the residual sample pulps for two high-grade and one low-grade gold assays, were reanalyzed 20 times using fire assay (25g charge). This was undertaken to assess the repeatability of the original reported assay result. Analyses for the three samples returned assay results that were consistent with the original reported gold assay result.

Framboidal pyrite aggregates (≤50mm in size) and laminations of pyrite (≤20mm widths) are common in the stratigraphic sequence, especially in black shale units. The framboidal pyrite is diagenetic and drill core assaying of this material returns grades <5ppb Au.

Other sulphides present at FGM in small quantities include galena, sphalerite and chalcopyrite, boulangerite (Pb5Sb4S11) and rarer still are tennantite (CuFe12As4S13), tetrahedrite (CuFe12Sb4S13), and bournonite (PbCuSbS3), which have been reported in processing plant sulphide concentrates (McArthur, 2012 and Townend, 2009)

Silver grades are low at Fosterville; usually about one tenth of the gold grade with only ~1% silver commonly in poured gold doré in the early years of sulphide gold operations. However, the silver content in poured doré has gradually increased to the present ~3% silver levels and may be related to the increase with depth of one or more accessory silver minerals, such as tennantite.

7.4 Controls on Primary Mineralization

At Fosterville primary gold mineralization is structurally controlled and localized by the discordant relationship between bedding and faulting (Figure 7-4). Gold mineralization is more continuous and of higher grades in fault segments where east-dipping beds occur adjacent to west-dipping footwall beds across faulting, such as along the Phoenix Fault (Boucher et al, 2008a), i.e. discordant-concordant structural setting (locally termed oblique/parallel or parallel/oblique). Mineralized shoots are typically 4m to 15m thick, 50m to 150m up/down dip and 300m to 1500m+ down plunge (Figure 7-8). Gold grades are relatively smoothly distributed with both extremely high values and extremely low values uncommon.

There are four geometric bedding-fault relationships present at Fosterville; primarily created through the interaction of west dipping faulting that links across fold closures, from an anticline in the west to a syncline in the east. The four bedding relationships across fault are locally referred to as parallel/parallel, parallel/oblique, oblique/oblique and oblique/parallel structural settings. These are briefly mentioned above in reference to the schematic cross section and further described below:

Parallel/Parallel (P/P) setting is where the bedding in the hangingwall and footwall is parallel (concordant) with faulting, such as where faulting becomes bedded to the west of footwall anticlines and east of a hangingwall syncline. Recent developments have shown that economic mineralization can form in parallel / parallel setting where the stress between slipping beds can form perpendicular stacked vein arrays to the bedding orientation, termed ladder veins. High grade visible gold and stibnite can also form in veins constrained by bedding units giving another mechanism for parallel / parallel mineralization

Parallel/Oblique (P/O) setting is where bedding hangingwall to faulting is parallel but the footwall bedding is at an oblique angle (discordant) to faulting. Parallel/oblique settings occur at Fosterville where a west dipping fault offsets a footwall anticline axial plane. This structural setting is generally well mineralized.

Oblique/Oblique (O/O) setting is where bedding in both the hangingwall and footwall is oblique to faulting. Oblique/oblique settings occur where a west dipping structure passes through east dipping stratigraphy between the hangingwall anticline and footwall syncline axial planes. This structural setting is variably mineralized.

Oblique/Parallel (O/P) setting is where bedding hangingwall to faulting is oblique to faulting and the footwall bedding is parallel. Oblique/parallel settings occur at Fosterville where a west dipping fault offsets a syncline axial plane. This setting is also generally well mineralized.

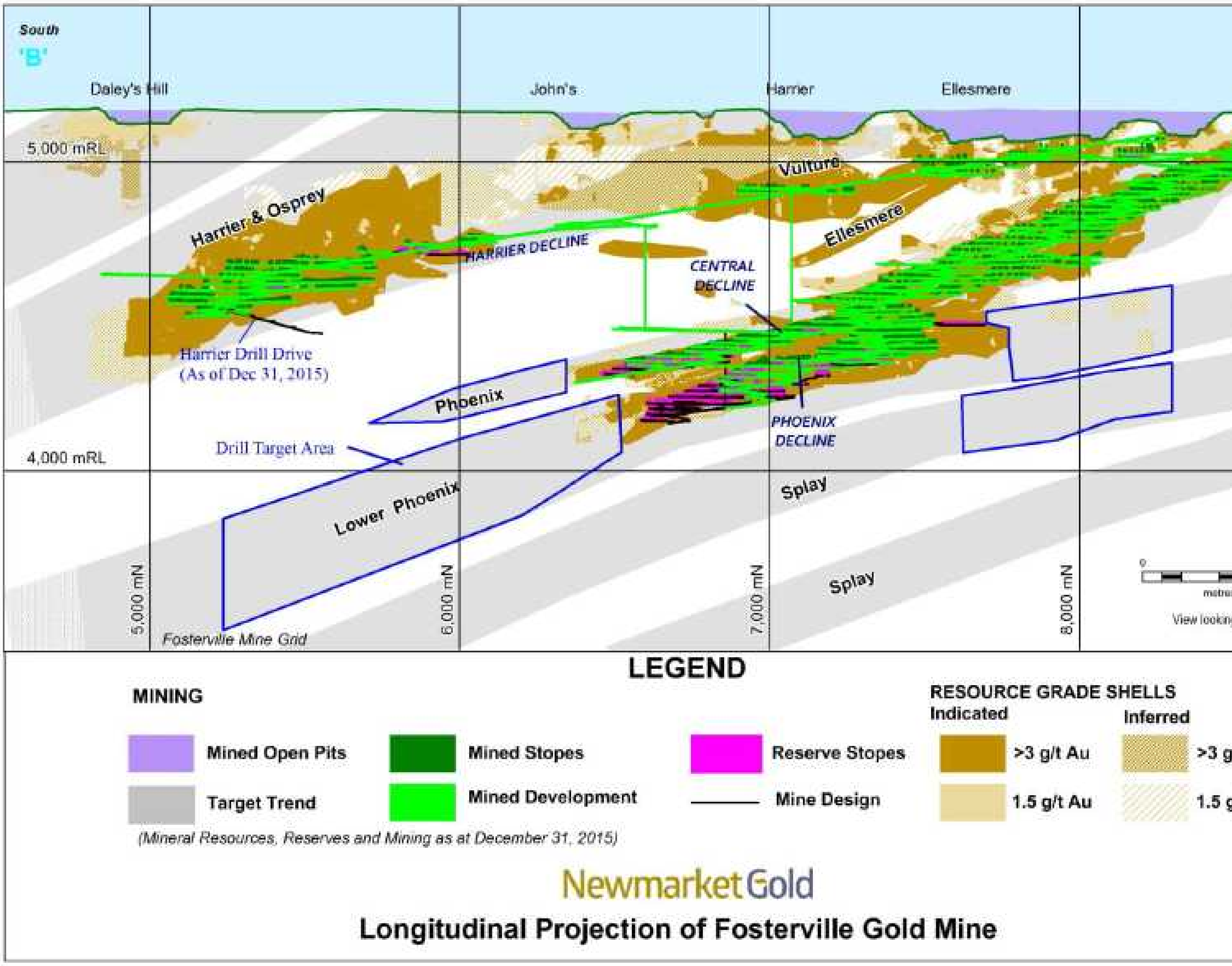


Figure 7-8 Fosterville Fault Zone Longitudinal Projection showing Resources, Reserves, mining and target areas.

7.4.1 Fosterville Fault Zone

The Fosterville Fault Area represents a linear trend of gold mineralization within MIN5404 extending from Daley's Hill in the south to Rehe's pit in the north (Figure 7-3). The trend is controlled by fault interplay between the Fosterville Anticline and the Phoenix Syncline, which form a parasitic fold conjugate in the west dipping limb of the regional synclinorium.

Early deformation of the sedimentary packages developed laminated quartz veins proximal to shale packages that were susceptible to accommodation flexural slip. As deformation intensified these preserved shale lamination became nucleation points for brittle fault failure across east dipping bedding. Fault activation was greatest around shale units that contained carbon forming graphitic surfaces to reduce slipping friction forces.

The accommodation of strain between the syncline and anticline provided a fault mesh where there is a complex interplay between east and west dipping faults. From 9000mN to 7500mN, this interplay was largely not recognized due to the short eastern limb length. Over this northing range most of the compressive force was accommodated by large fault offsets of the Fosterville and Phoenix Faults. As exploration continued south, the syncline and anticline appeared to diverge subtly from one another increasing the eastern limb length. Force accommodation between the zones had longer distances to cut across resulting in faulting that has reduced measurable offset.

The result of the reduced offset appears to have an effect on how the east dipping rocks accommodated faulting at depth, with ladder veins systems opening in bedding parallel zones along shale boundaries. Fluids utilizing these pathways were not constrained to one pathway as seen in the Phoenix and Falcon Zones, but use a diverse network including hinges, sedimentary units, east and west dipping faults. There also appears to be an element of fluid pressurization injecting up plunge seeking lower pressure environments.

This fluid pressurization appears to be strongly coincident with the increase of massive quartz, quartz stibnite and visible gold at depth. Quartz zones can be several meters thick and suggest an element of hydraulic fracturing as fluid pressure overcomes lithostastic pressure and injects into zones, which act as a trap. This occurs in the Eagle / Lower Phoenix interaction zone around the Fosterville anticline. The orientation of the zone is strongly influenced by the late stage north south compression that gives the Fosterville Fault Zone orebodies their 20^{o} plunge to the south

Midway along the mineralized trend at approximately 8800mN (Falcon Pit area), a fold culmination (dome) occurs. The culmination causes plunge reversals to both folds and mineralization, and to the north of the culmination, the footwall syncline and mineralization shoots plunge gently to the north. Similarly, south of the culmination, the footwall syncline and mineralization shoots plunge gently to the south.

The Fosterville Fault Zone consists of 9 primary and 8 secondary Mineralization Zones including:

<u>Primary</u>

- Phoenix
- Falcon
- Harrier
- Lower Phoenix
- Lower Phoenix Footwall
- Ellesmere
- Vulture
- Osprey
- Robin
- Eagle
- East Dippers
- Allwood
- Kestrel

<u>Secondary</u>

- Splays
- Raven
- Shamrock
- Griffon

7.4.2 Robbin' s Hill Area

Although the Robbin's Hill area (Figure 7-3) is less intensely explored at depths below 100m, fault-controlled primary gold mineralization exhibits structural and lithological controls similar to the Fosterville Fault Zone. Faults are clearly the primary control on mineralization with lithology providing secondary control. However, the Robbin's Hill Area is more structurally complex, with multiple small scale offsets of fold closures, creating smaller zones of discordant bedding relationships and stacked lenses of gold mineralization. The mineralized faults generally strike within 20° of north and dip moderately to steeply west.

Rhyolitic dyke associated gold mineralization also occurs in the area, with mineralization mainly within 2m of the dyke contacts. The rhyolitic porphyry dyke bodies have a general north-south trend, are typically subvertically orientated and are observed to often intrude anticlinal axial planes.

Higher grade gold zones are controlled by the intersection of fault controlled mineralization with the dykes.

7.5 Controls on Oxide Mineralization

Minor re-mobilization of gold into the immediately surrounding country rocks has resulted in an approximate 50% increase in the width of mineralization and consequent reduction in gold grade. There is no evidence of a wide spread high-grade supergene zone immediately below the water table.

Other elements have been more significantly affected by weathering processes. Dissolution of sulphur by oxidizing groundwater above the water table has effectively removed all sulphur from the oxide zone. Arsenic has been strongly remobilized over a zone five to ten times the width of mineralization. The greater width of anomalous arsenic values in the oxide zones makes arsenic soil geochemistry a very useful tool for finding exposed gold mineralization.

Geochemical studies (Arne and House, 2009) found evidence of Fe or Mn oxide minerals scavenging Au, As or Sb in the weathered zone and that raw concentrations of Au, As and Sb may be used for defining secondary dispersion (with allowance made for the rock type for Sb).

8 DEPOSITTYPES

Gold mineralization at Fosterville is relatively homogenous with only one deposit type present. There are minor variations in the host rock type and structural setting. Fosterville-type deposits form a sub-group of orogenic gold deposits that are typified by gold occurring in fine grained arsenopyrite and/or pyrite disseminated in country rocks as a selvage to faults or veins. Fosterville-type deposits and classic vein-hosted deposits are effectively end members with many orogenic gold deposits displaying features of both.

Primary mineralization at Fosterville is controlled by late brittle faulting. These late brittle faults are stacked, generally steeply west dipping with reverse movement varying from a few meters to over 150m. In the upper parts the fault system a series of moderately west dipping reverse splay faults occurs in the footwall of the Fosterville Fault. Primary gold mineralization occurs as disseminated arsenopyrite and pyrite forming as a selvage to veins in quartz - carbonate veinlet stockwork. The mineralization is structurally controlled with high-grade zones localized by the geometric relationship between bedding and faulting. Mineralized shoots are typically 4m to 15m thick, 50m to 150m up/down dip and 300m to 1,500m+ down plunge. These sulphide bodies are the primary target for exploration activities, especially where there is potential for grades in excess of 3 g/t Au (i.e. above underground cut-off gold grades).

Much less significant is the presence of primary native gold that where observed, is spatially associated with stibnite and quartz. This stibnite-quartz mineralization occurs as a late overprint/replacement of quartz-carbonate within brittle faulting as described above, and is most commonly associated with the Phoenix and Lower Phoenix Footwall and footwall east dipping faults.

Within the oxide zone, there has typically been minor re-mobilization of gold into the immediately surrounding country rocks which has resulted in an approximately 50% increase in the width of mineralization and consequent reduction in gold grade. There is no evidence of a wide spread high-grade supergene zone immediately below the water table. There is no current focus on exploring for additional oxide resources.

9 EXPLORATION

9.1 Pre-1992 Exploration

Modern exploration commenced at Fosterville during the 1970' s. Apollo International Minerals NL drilled three HQ diamond holes in what is now the Hunts area. Noranda Inc. drilled three HQ diamond holes in the Daley' s Hill area. None of these holes have been included in the drilling database due to uncertainty in their collar locations.

From 1987 to 1991 Bendigo Gold Associates and later Brunswick drilled 488 RC holes and six HQ diamond holes targeting oxide mineralization on the Fosterville Fault and the Robbins Hill area. This program resulted in the development of a heap leach operation which commenced in 1991.

Brunswick also completed a 100m by 20m soil geochemistry grid across the project area and as far west as the Sugarloaf Range. The soil geochemistry was very effective at defining gold mineralization except where alluvial cover exceeded about two meters. Two preliminary IP/resistivity lines were completed with mixed results.

9.2 1992-2001 Exploration

On acquiring the Fosterville Mine Lease in 1992, Perseverance engaged a drilling contractor and started RC drilling for further oxide resources and reserves using a combination of cross over and face sampling hammers.

In late 1994, while continuing to explore for oxide mineralization, Perseverance began to drill for sulphide mineralization on the Fosterville Fault, potentially amenable to open cut mining. This drilling was almost entirely RC using a face sampling hammer with minor diamond drilling for metallurgical and geotechnical purposes and extended from 6000mN to 10700mN. Most of the drilling was completed by 1997 with minor infill drilling continuing to 1999.

Section spacing was either 25m or 20m except in two small zones in the Falcon and Ellesmere areas where 12.5m sections were drilled. This drilling program was generally restricted to within 100m of surface, extending to a vertical depth of 150m below surface in the Central North area, reflecting the perceived limits of open cut mining. The data from this drilling program formed the basis of the 1997 Sulphide Project Feasibility Study which was later updated in 2000 (Perseverance, 1997; 2000).

Two deep diamond holes, SPD7 and SPD8 were also drilled. SPD7 was drilled beneath the Central Ellesmere pit and intersected 53.8m at 1.97 g/t Au (drill hole abandoned in mineralization) from 382m, while SPD8 was drilled to 450m below Central North intersecting only 2.0m at 0.58 g/t Au on a splay fault some 60m to the east of the Fosterville Fault. A 25m by 25m gradient array IP survey was conducted in the Robbins Hill area in 1997. This survey did not conclusively define gold mineralization; however, it was successful in mapping carbonaceous shales and alluvial channels.

9.3 2001-2008 Exploration

The drilling program on the Fosterville Fault Zone commenced in July 2001 and is ongoing. For the majority of this period, the surface drilling activities were conducted by Silver City Drilling Pty Ltd (drilling contractor) and the underground drilling activities by Deepcore Pty Ltd (drilling contractor). Resource definition holes are usually drilled with RC pre-collars and NQ2 diamond tails. The sectional spacing ranges from 200m to 50m with the vertical spacing of intersections usually 50 metres.

A small number of RC-only holes were drilled where the target was shallow and exploratory. Once definitive targets were defined by this type of drilling, the drilling methods changed to those used for resource definition drilling described above.

The change in drilling methods to largely oriented diamond core, intensive re-mapping of old oxide pits and a change in logging methods to collect detailed grain size data allowing sequence stratigraphic analysis allowed much more detailed and robust geological models. These geological models allowed a better understanding of the controls on gold mineralization which in turn resulted in the better targeting and more efficient use of drilling.

The post-2001 exploration resulted in the discovery and definition of the Phoenix, Wirrawilla and Farley' s deep zones. In addition the Falcon, Ellesmere and Harrier Zones were extended. Modest additions to resources were made at the Daley' s Hill, Sharkey' s and Hunts Deposits.

Two further IP/resistivity surveys were completed in 2001 and 2005. The 2001 survey consisted of four lines of 50m node spacing over the Central Area. This survey was designed to define gold mineralization at depths of between 50m to 250m. The data was inverted to make a model in real space. Anomalies were defined along the Fosterville Fault Zone, but the 50m node spacing meant that the survey resolution was unable to distinguish the carbonaceous shale in the hangingwall of the Fosterville Fault from mineralization in the footwall of the Fosterville Fault. In 2005 another four IP lines were completed across the northern end of the Fosterville Goldfield, covering the Sugarloaf geochemical anomaly, the Fosterville Fault Zone and the Robbin' s Hill Area. This survey defined weak anomalies over the Sugarloaf geochemical anomaly and the strike projection of the Fosterville Fault Zone north of MIN5404.

9.4 2008-2011 Exploration

The drilling programs at Fosterville have essentially been continuous from 2001 to 2011. Most of the surface drilling was conducted by Silver City Drilling Pty Ltd until November 2009 and thereafter by Macquarie Drilling (drilling contractor). Deepcore Pty Ltd has continued to provide all underground diamond drilling services as well as completing diamond holes from surface. Depending on depth to drill target the surface resource definition drilling varies in drilling method from holes with RC pre-collars and NQ2 diamond tails to holes exclusively using diamond drill methods of HQ and/or NQ2 sizes.

The 2008 surface diamond drilling program tested the characteristics and extent of resources of the Wirrawilla (renamed as Harrier UG) and Phoenix resource areas. Thirty six holes totaling 16,253m were completed with 86% completed in Harrier UG area and 14% in the Phoenix area.

The program resulted in the discovery of extensions to three north striking, west dipping areas of gold mineralization within the Harrier UG area: the Osprey; Raptor; and, Harrier Base Fault zones. The zones are situated 1.5km south of the current Phoenix Mineralized Zone and are interpreted to be at a higher stratigraphic level, but down plunge of the Harrier open-pit Mineralized Zone, which was mined in 2007.

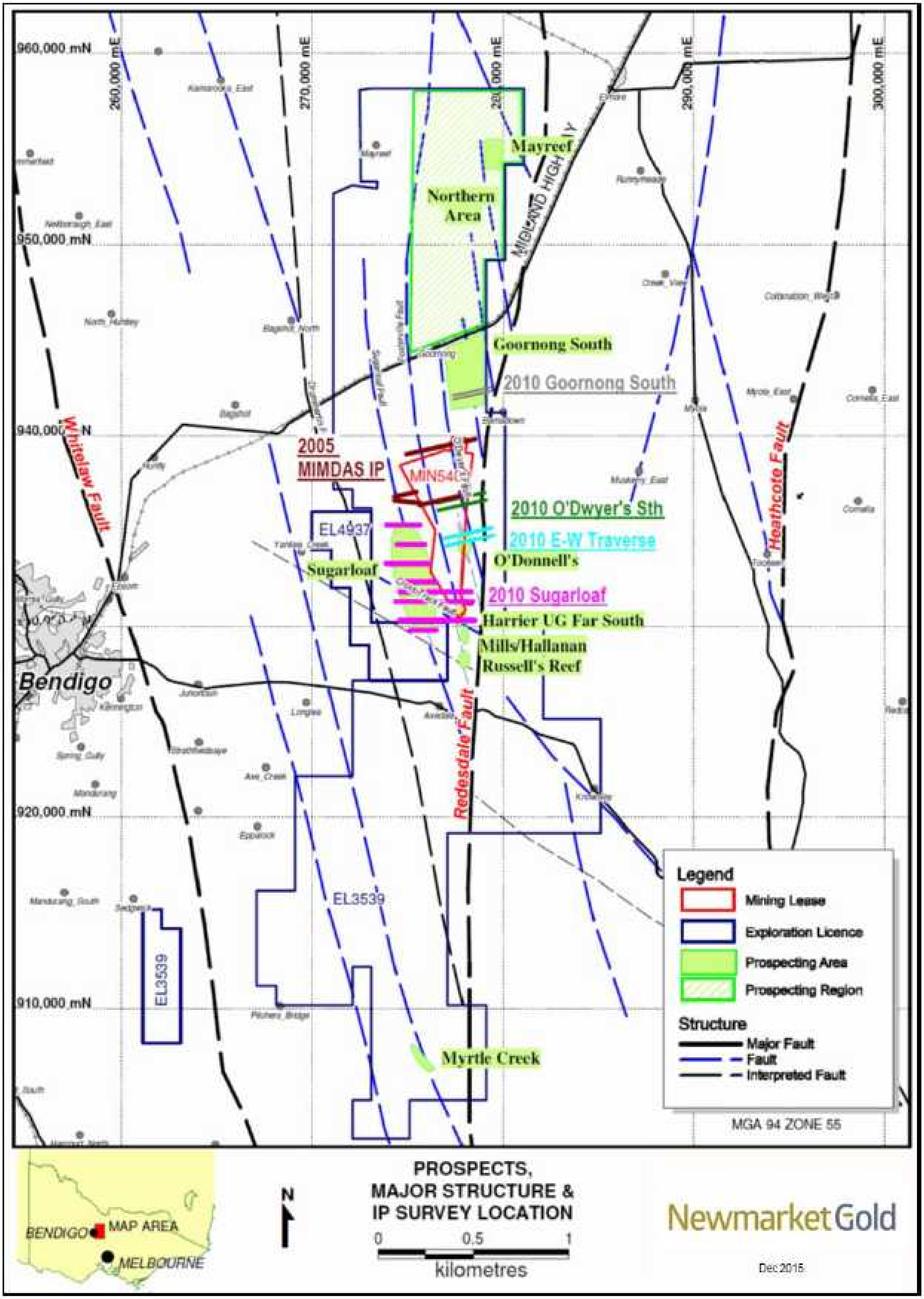
The 2009 exploration program consisted of an additional 12,179m of drilling that served as the basis for an underground resource estimate in Harrier using a 3.0 g/t Au lower cut-off.

Additional exploration drilling in 2009 consisted of 6,633m of drilling on Phoenix Extension, 1,051m on other targets on the Fosterville Mine Lease as well as 1,695m in ten holes on the Myrtle Creek Prospect (EL3539) south of the FGM.

IP/resistivity works were conducted that highlighted areas for targeting, as shown in Figure 9-1 and published in the 2015 Technical Report for Fosterville Gold Mine.

The 2010 exploration program consisted of 49,980m of drilling; the majority of which was directed towards the Harrier (47%) and Phoenix (30%) Zones to both extend zones and reduce drill spacing to upgrade the confidence in the resources prior to reserve studies. The balance of the exploration was directed to other targets on the Mine Lease and a small amount of drilling was undertaken on the exploration tenements surrounding the Mine Lease.

The 2011 exploration program consisted of 17,032m of drilling directed towards thirteen different target areas on the Mining Lease, some of which are push backs on existing open pits and others are underground mining target areas.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290,000 mE
300,000 mE
Mayreef
Northern Area
MIDLAND HIGHWAY
Goornong South
2010 Goornong South
2005 MIMDAS IP
MIN5404
2010 O'Dwyer's Sth
2010 E-W Traverse
O'Donnell's
EL4937
Sugarloaf
2010 Sugarloaf
Harrier UG Far South
Mills/Hallanan
Russell's Reef
Whitelaw Fault
Heathcote Fault
Redesdale Fault
Bendigo
EL3539
Myrtle Creek
Legend
Mining Lease
Exploration Licence
Prospecting Area
Prospecting Region
Structure
Major Fault
Fault
Interpreted Fault
MGA 94 ZONE 55
BENDIGO
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PROSPECTS, MAJOR STRUCTURE & IP SURVEY LOCATION
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kilometres
Newmarket Gold
Dec 2015

Figure 9-1 Plan of IP areas and Prospects surrounding Fosterville Gold Mine.

9.5 2012 to present Exploration

2012 Exploration

During 2012, exploration drilling of 13,741m at Fosterville was focused upon four main target areas within the Mining Lease: the Robbin's Hill and Falcon North areas (drilled from surface) and southern extensions of the Harrier and Phoenix Mineralized Zones (both drilled from underground platforms)

Drill results at Falcon North were disappointing, but positive results were returned for drilling beneath historical oxide pits at Robbin's Hill and O'Dwyer's South (the O'Dwyer's South pit was subsequently deepened in 2012). Underground extension drilling 100m south of the Harrier UG resource was successful in intersecting several moderate grade gold intercepts, sufficient for follow up drilling in 2013.

Phoenix Extension drilling, 100m south along strike (down plunge) from the Phoenix resource intersected significant gold mineralization over a 60-metre down dip length and is associated with the Phoenix Fault recognized another 100m south in previous drilling where it is also strongly mineralized. In total there are five significant Phoenix drill intercepts (greater than 30 gram-meters) recorded for this area (6500mN to 6750N) and collectively outline a 250 metre strike length with substantial down-plunge exploration potential.

2013 Exploration

In 2013 exploration drilling (14,925m) focused on several extension drilling programs, including along strike (down plunge) programs on Harrier UG, Phoenix and Lower Phoenix Zones. The Lower Phoenix target was also tested to the north along strike (up plunge) for gold potential.

At Harrier UG a section of drilling (4850N) tested 100m south of existing drill data, and although the program was successful in identifying moderate to high-gold grades the width of intercepts is generally less than 3m and further drilling is not planned south of 4850N to target Harrier gold mineralization.

Progressive drilling and geological modeling of the Phoenix Fault System in 2013 confirmed the presence of two favorable structural zones (Phoenix and Lower Phoenix) below the 4500mRL, which host significant gold mineralization. Both the Phoenix and Lower Phoenix Zones are proximal to fold axial plane offsets along the Phoenix Fault System. In total there were approximately 55 drill intercepts that record >30 gram-meter values for the Phoenix and Lower Phoenix target areas.

Definition drilling into the Phoenix structure during 2013 also reaffirmed mineralization continuity of the known Mineral Resources in this area. However, drilling is absent on the Phoenix structure south of 6500mN for a distance of 450 meters in strike length

2014 Exploration

In 2014 exploration diamond drilling at Fosterville saw 138 drill holes completed for a total of 31,550.4 meters. The drilling programs continued to focus on the Lower Phoenix and Phoenix targets with 14 holes reporting high-grade gold intercepts. The results of the 2014 drilling were fully reported on February 2, 2015 in a Crocodile Gold News Release.

The holes drilled in 2014 establish the continuation of gold mineralization down-plunge and identified multiple mineralized faults below the Phoenix and Lower Phoenix Faults. Drill results for three mineralized faults, namely the Lower Phoenix Footwall, East Dipping and Kestrel Faults, have the potential to extend Mineral Resources along strike.

2015 Exploration

Exploration activities in 2015 were focused on near-mine targets to replace reserves by extending known ore shoots. Exploration ventures continue to strive to locate and define new ore shoots of economic gold mineralization within close proximity to current underground workings and future open pit projects.

Exploration diamond drilling during 2015 consisted of 196 holes that totaled 42,700m, and was focused upon five main target areas within the Mining Lease: southern and northern extensions of the Lower Phoenix and Fosterville Splays (drilled from surface and underground), the southern extensions of the Lower Phoenix Footwall, Phoenix, Eagle and Kestrel Fault systems.

Key Intercepts of the 2015 drilling include:

Eagle Fault intercepts (Figure 9-2):

- 386 g/t Au over 9.15m (ETW 3.35m) in hole UDH1238 ([1])
  - Including 5,283 g/t Au over 0.6m
- 268 g/t Au over 7.85m (ETW 2.77m) in hole UDH1255([1])
  - Including 5,276 g/t Au over 0.35m
- 16.38 g/t Au over 18m (ETW 16.56m) in hole UDH1364([1])
  - Including 145.3 g/t Au over 0.95m
- 100 g/t Au over 2.8m (ETW 2.64m) in hole UDH1390([1])
  - Including 266 g/t Au over 0.6m
- 161 g/t Au over 7.35m (ETW 4.94m) in hole UDH1481([1])
  - Including 499 g/t Au over 2.25m

East Dipping Fault intercepts, which confirm high-grade gold mineralization on east dipping structures (Figure 9-3):

- 34.47 g/t Au over 5.10m (ETW 4.09m) in hole UDH1294([1])
  - Including 158 g/t Au over 0.95m
- 15.84 g/t Au over 5.2m (ETW 4.86m) in hole UDH1374([1])
  - Including 1,321 g/t Au over 0.35m
- 45.47 g/t Au over 14.95m (ETW 13.5m) in hole UDH1408([1])
  - Including 1,321 g/t Au over 0.35m
- 645 g/t Au over 3.5m (ETW 3.41m) in hole UDH1456([1])
  - Including 7,368 g/t Au over 0.3m

Kestrel Structure intercepts (Figure 9-4 & Figure 9-5):

- 6.16 g/t Au over 11.05m (ETW 5.12m) in hole UDH1122
- 7.27 g/t Au over 6.1m (ETW 6.04m) in hole UDH1372
- 9.77 g/t Au over 4.55m (ETW 3.95m) in hole UDH1406

Lower Phoenix Fault intercepts (Figure 9-6):

- 6.68 g/t Au over 5.4m (ETW 4.5m) in hole UDH1433 and;

- 16.53 g/t Au over 4.5m (ETW 3.99m) in hole UDH1444

Lower Phoenix Footwall Fault intercepts (Figure 9-6):

- 77.87 g/t Au over 6.30m (ETW 4.33m) in hole UDH1219A([1])
    - Including 332 g/t Au over 0.27m
- 7.25 g/t Au over 12.50m (ETW 7.63m) in hole UDH1240
- 159 g/t Au over 7.55m (ETW 4.09m) in hole UDH1365([1])
    - Including 1,290 g/t Au over 0.46m)
- 5.94 g/t Au over 7.4m (ETW 6.48m) in hole UDH1366

Phoenix Fault intercepts (Figure 9-6):

- 7.49 g/t Au over 5.05m (ETW 4.59m) in hole UDH1016
- 7.91 g/t Au over 6.9m (ETW 5.07m) in the hole UDH1303
- 7.65 g/t Au over 4.6m (ETW 4.13m) in hole UDH1439

Notes:

(1) Visible gold present in drill intercept ETW - Estimated true width

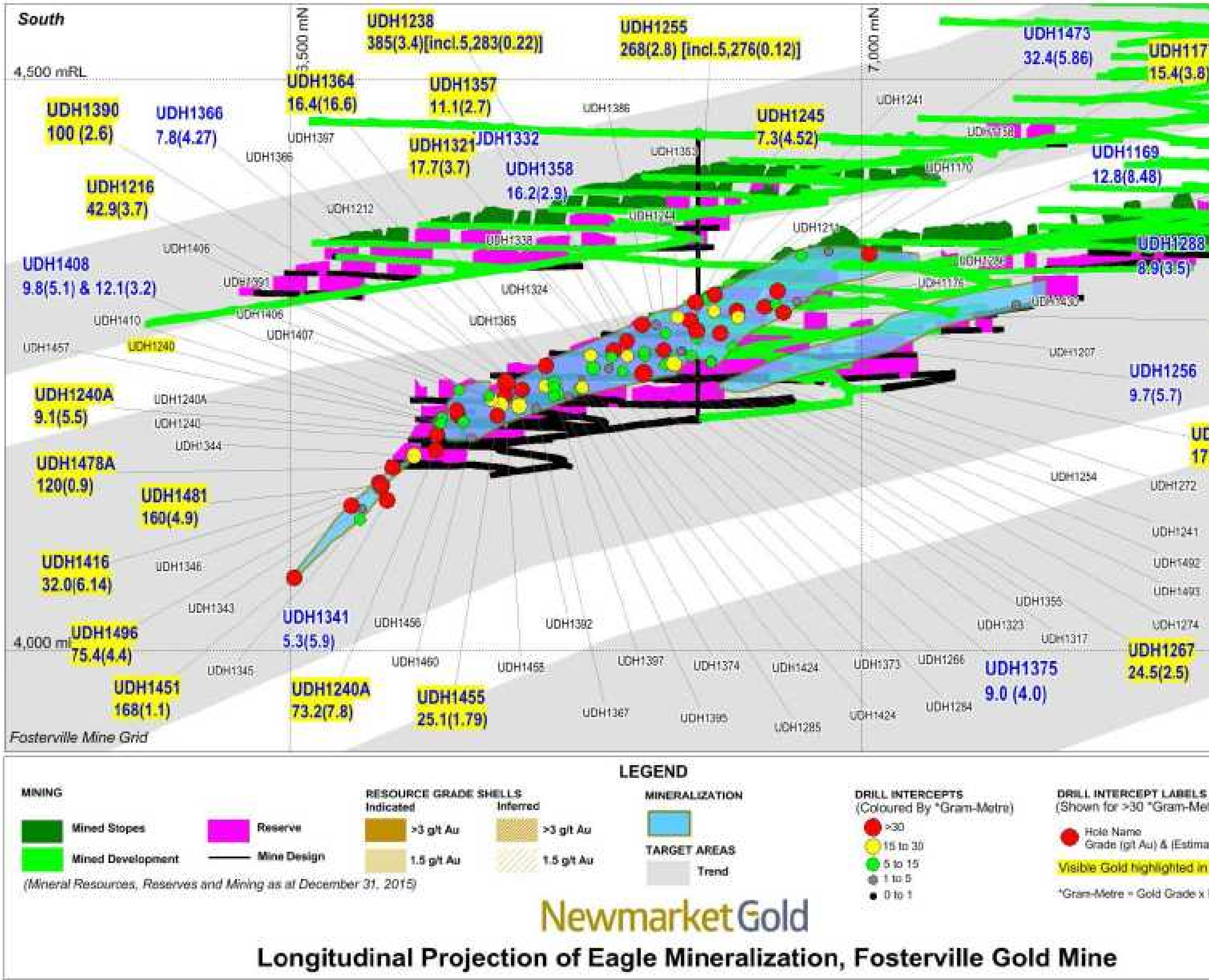

Figure9-2 Longitudinal Projection of Eagle Fault Mineralization

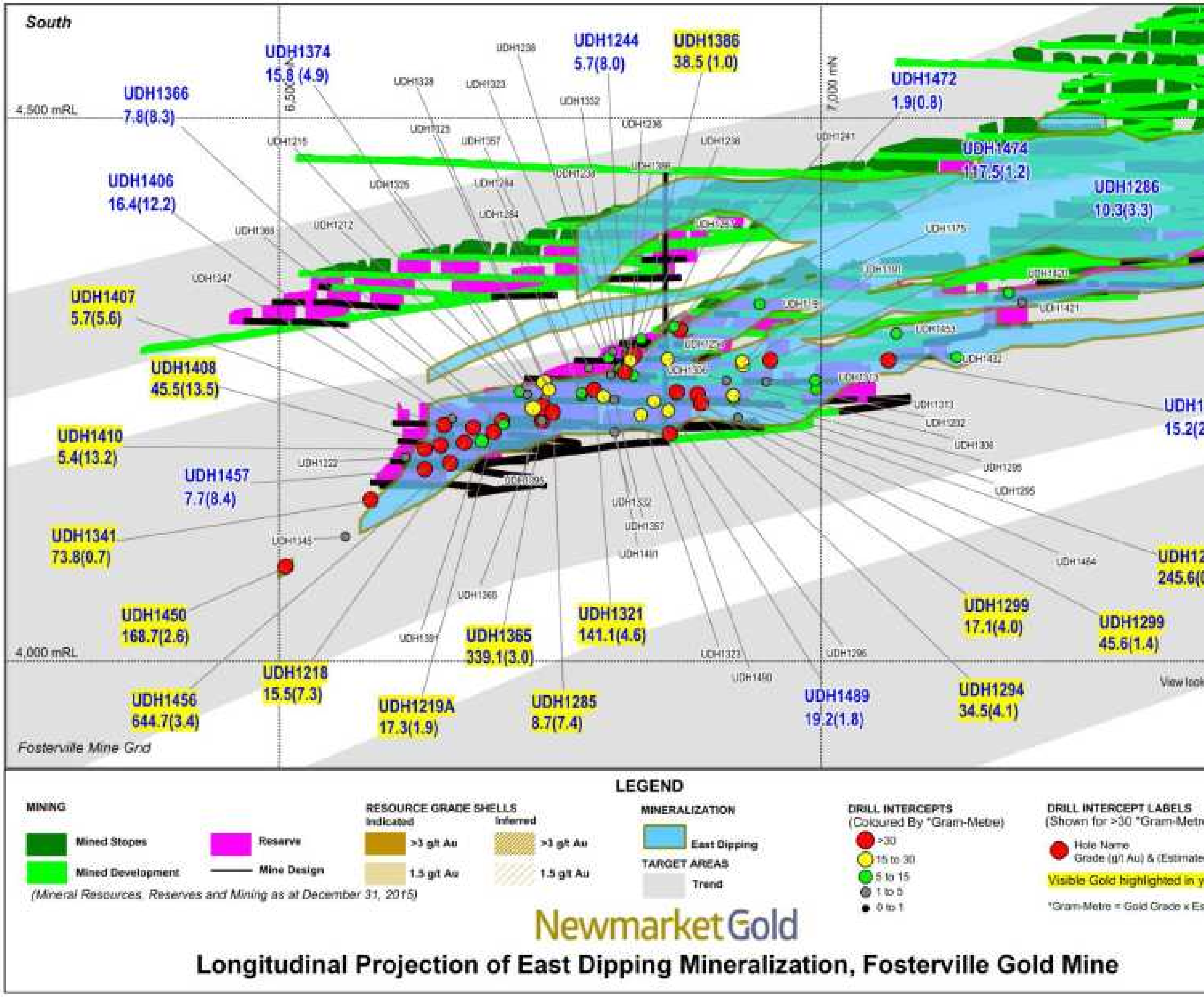

Figure9-3 Longitudinal Projection of East Dipping Mineralization

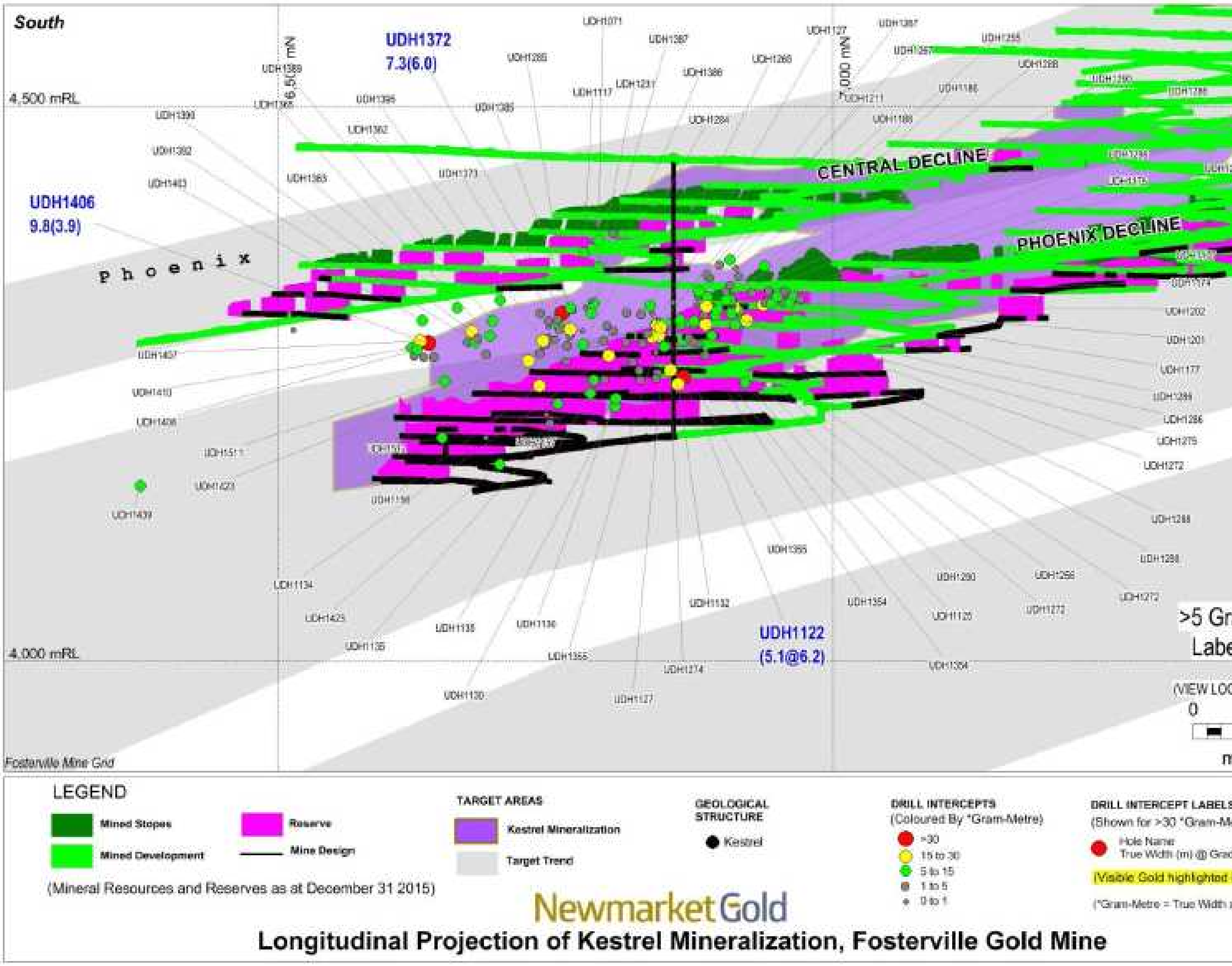

Figure9-4 Longitudinal Projection of Kestrel Structure Mineralization (>5 gram-metreAu)

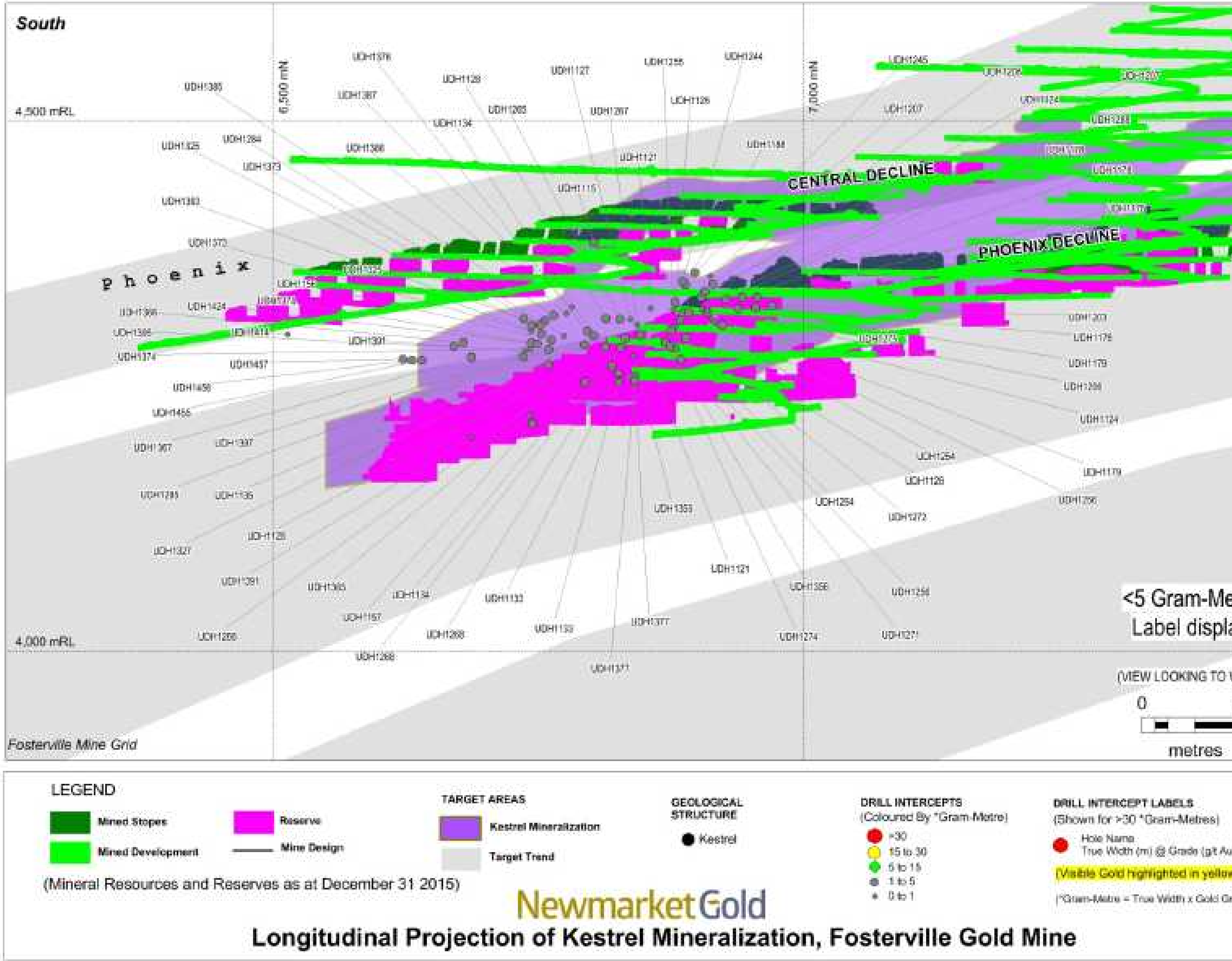

Figure9-5 Longitudinal Projection of Kestrel Structure Mineralization (<5 gram-metreAu)

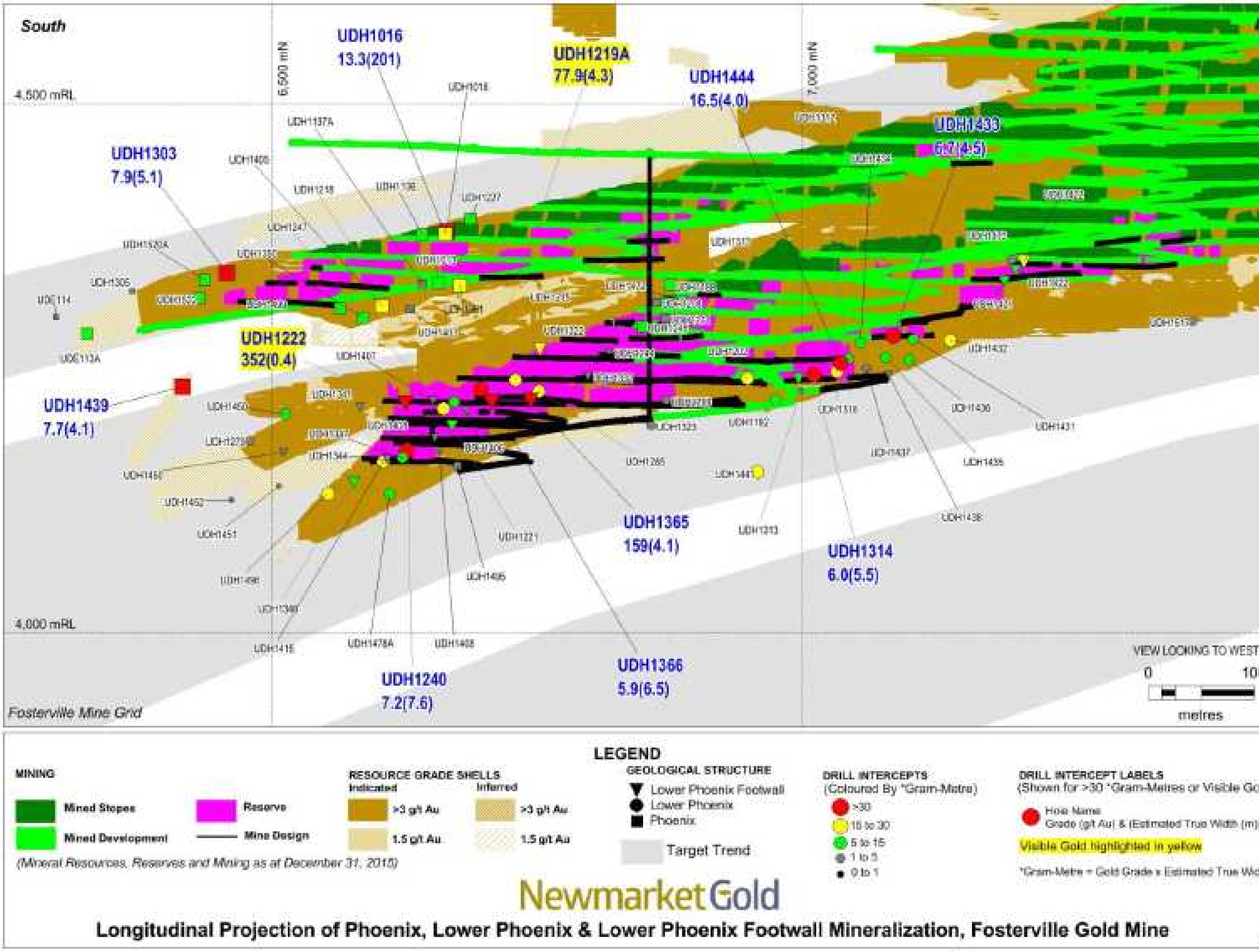

Figure 9-6 Longitudinal Projection of Phoenix, Lower Phoenix & Lower Phoenix Footwall Mineralization

2016 Exploration

The proposed exploration activities in the current year are focused on near-mine targets and extending currently known resources within MIN5404. However, in subsequent years, exploration is planned for target areas away from the Fosterville Mine Lease within the north-south extensive EL3539. The intent of the exploration is to replace reserves at Fosterville by extending presently known ore shoots, but to also locate and define new centers of economic gold mineralization applicable to open pit or underground extraction. Near mine exploration activities for 2016 include:

Northern Lower Phoenix (7850mN & 8050mN)

Collectively these two programs target the up-plunge extension of the Lower Phoenix gold system and will also support the gold results that exist for the northings 8200mN and 7700mN. Initial interpretations of the7700mN and 8200mN drill sections indicate that gold mineralization may continue between them. Gold mineralization present on 8200mN includes 4.7 g/t Au over 7.9m in hole UDE090 and 6.5 g/t Au over 6.0m in hole UDE091 and on the 7700mN section includes 3.2 g/t Au over 6.5m in hole SPD609A and 2.2 g/t Au over 5.3m in hole SPD609.

The 7850mN and 8050mN will commence in 2016. The total meterage planned for these programs, inclusive of Fosterville Splays targets on the same sections is 6,955m for a total cost of AUD $1.95M.

Southern Lower Phoenix & Lower Phoenix Footwall (6200mN)

This program commenced in 2015 and will carry into 2016. It targets the down-plunge extension of the Lower Phoenix gold system, 200m south of existing mineral resources, and 370m south of existing mineral reserves. No drilling exists south of the 6400mN that tests the Lower Phoenix gold system. The 6200mN program will provide an increased understanding of the continuity of the Southern Lower Phoenix gold system, and provide guidance on future resource definition drilling and mine development planning. The total planned meters to complete this program in 2016 is 435m for a total cost of AUD $0.1M.

Phoenix, Lower Phoenix, Lower Phoenix Footwall (5450mN)

This program will provide geological knowledge to the south of the Lower Phoenix gold system. The 5450mN underground diamond drill program targets approximately 1000m down plunge of the Phoenix and Lower Phoenix gold systems. An understanding of the position of the syncline and anticline will also provide valuable structural information necessary for future exploration targeting. As part of this program, capital development is required to provide an ideal drill platform. This development commenced late in 2015 and is anticipated to be completed by mid-2016. Preliminary designs for the underground diamond drill program consist of three parent holes with eight daughter holes totaling 6,610m and will cost approximately AUD $1.3M. Gold results and structural observations from the Southern Lower Phoenix drilling (6200mN) that commenced in 2015 will also guide drill hole planning.

9.6 Future Exploration Targets in EL3539

9.6.1 Goornong South

The Goornong South Prospect is located approximately 4km north of the Fosterville Mine Lease, where Fosterville style gold mineralization occurs beneath transported cover on privately owned land. The gold prospect was discovered by Perseverance during regional exploration in the mid 1990's. PSV identified a 1.3km long anomalous zone of gold mineralization and systematically drilled the anomaly between 1995 and 1999 for its open pit potential. The drilling comprises 71 RC holes (totaling 4,482m) and one diamond hole (69m) with a further eight aircore holes (293m) drilled for ground water monitoring purposes.

Perseverance subsequently reported a Historic Resource in their 1999 Annual Report as shown in Table 9-1. However, Newmarket Gold is not treating the Historical Resource as a current mineral resource as a QP has not done sufficient work to classify the Historic Resource, or comment on the reliability of the estimate.

Table 9-1 Historic Resource of the Goornong South Prospect Perserverance (1999)

Historical Mineral Resource (PSV 1999) - Goornong South Prospect									
Classification	Measured			Indicated			Inferred		
	Tonnes (kt)	Grade (g/t Au)	Insitu Gold (Oz)	Tonnes (kt)	Grade (g/t Au)	Insitu Gold (Oz)	Tonnes (kt)	Grade (g/t Au)	Insitu Gold (Oz)
Oxide	216	1.3	9,300	535	1.3	23,100	32	1.6	1,700
Sulphide (High Grade)	7	1.7	400	46	1.6	2,400	373	1.5	18,200
Sulphide (Low Grade)	3	0.7	100	11	0.7	300	140	0.8	3,700
Total Sulphide	10	1.4	500	57	1.4	2,700	513	1.3	21,800
Total Oxide & Sulphide	226	1.3	9,800	592	1.4	25,800	545	1.3	23,500

Notes:

Historical Resource as reported in Perseverance Annual Report 1999.

Newmarket Gold is not treating the historical estimate as a current mineral resource as a QP has not done sufficient work to classify the historical estimate or comment the reliability of the estimate.

Reporting lower cut-off gold grades used are ≥0.5g/t Au for oxide, 0.5-1.0 g/t Au for sulphide low-grade and >1.0 g/t Au for sulphide high- grade.

Bulk Density values set to 1.8t/m³ for clay, 2.4t/m³ for oxide and 2.8t/m³ for sulphide materials.

Resource block grades estimated by Ordinary Kriging of 50m spaced drill sections.

Mineral Resources have been rounded to 1,000 tonnes, 0.1 g/t Au and 100 ounces. Minor discrepancies in summation may occur due to rounding.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

In 2010 Northgate reviewed the Goornong South area for its potential to host gold mineralization amenable to underground mine extraction. The initial exploration saw completion of two lines of ground IP/resistivity survey (Figure 9-1) to the south of the prospect in order to identify chargeability anomalies along strike from the sulphide mineralization at Goornong South. Chargeability anomalies were encountered on both IP lines and a five diamond drillhole program (totaling 1,532m) was duly completed.

The 2010 diamond drilling program was undertaken from the roadside and spans about a 750m north-south trend. The program was unfortunately cut short owing to fiscal constraints at that time and the IP anomalies were not drill tested. However, of the five holes drilled, three returned assay intercepts averaging greater than 2.5 g/t Au and the strike length of the prospect has been extended southwards a further 300m.

Structural measurements of the available drill core indicate that the folding of the Ordovician turbidites plunges southwards at <20° and it is inferred that gold mineralization will also have this plunge analogous to structural controls of mineralization at Fosterville.

Newmarket Gold recommends a review of the Goornong South prospect as a potential underground Fosterville-style gold occurrence.

9.6.2 Hallanan's

The Hallanan's Prospect area, located 1 km south of the Fosterville Mine Lease, was explored for oxide gold by Perseverance between 1994 and 1998. During this period Perseverance completed 104 RC drill holes (totaling 6,245m with an average drill hole length of 60m), two diamond holes (109m) and 11 monitoring bores (354m). Gold mineralization was identified in drill intercepts over a 750m north-south trend and at the end of drilling a Historic Resource was estimated and reported by Perseverance in their 1999 Annual Report as shown in the following table (Table 9-2). However, Newmarket Gold is not treating the Historic Resource as a current mineral resource as a QP has not done sufficient work to classify the Historic Resource, or comment the reliability of the estimate

Table 9-2 Historical (1999) Perseverance Mineral Resource of Hallanan's Prospect

Historical Mineral Resource (PSV 1999) - Hallanan's Prospect									
	Measured			Indicated			Inferred		
Classification	Tonnes (kt)	Grade (g/t Au)	Insitu Gold (Oz)	Tonnes (kt)	Grade (g/t Au)	Insitu Gold (Oz)	Tonnes (kt)	Grade (g/t Au)	Insitu Gold (Oz)
Oxide	281	1.4	12,900	169	1.4	7,600	41	1.2	1,600
			-			-			-
Sulphide (High Grade)	89	1.5	4,400	240	1.5	11,500	521	1.7	28,600
Sulphide (Low Grade)	35	0.8	900	66	0.8	1,600	124	0.8	3,000
Total Sulphide	124	1.3	5,200	306	1.3	13,100	645	1.5	31,700
			-			-			-
Total Oxide & Sulphide	405	1.4	18,100	475	1.4	20,700	686	1.5	33,300

Notes:

Historic Resource as reported in Perseverance Annual Report 1999

Newmarket Gold is not treating the historical estimate as a current mineral resource as a QP has not done sufficient work to classify the historical estimate or comment the reliability of the estimate.

Reporting Lower cut-off gold grades used are ≥0.5 g/t Au for oxide, 0.5-1.0 g/t Au for sulphide low-grade and >1.0g/t Au for sulphide high-grade.

Bulk Density values of 1.8t/m³ for clay, 2.4t/m³ for oxide and 2.8t/m³ for sulphide materials.

Resource block grades estimated by Ordinary Kriging of 25 & 50 m spaced drill sections.

Mineral Resources have been rounded to 1,000 tonnes, 0.1 g/t Au and 100 ounces. Minor discrepancies in summation may occur due to rounding.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

No exploration activity has been undertaken on the Hallanan's Prospect since 1999 and during the intervening period to 2012 much has been learnt about structural controls of Fosterville-style gold mineralization at the nearby Mine Lease. Diamond drill core is virtually absent from the Hallanan's Prospect, and this coupled with an absence of any deep drilling, with RC drilling only averaging 60m in depth, the prospect is viewed by Newmarket Gold as being under explored for underground gold targets. The area is to be reviewed for drill testing in the future.

9.6.3 Harrier UG Far South

The Harrier UG Far South Prospect is conceptual in nature being the along strike/down plunge extension of the Harrier UG resource located in the southern portion of the Fosterville Mine Lease. Underground diamond drilling at Harrier South (4850mN, Mine Grid) was completed in 2013 from within EL3539. This drilling is to be reviewed in conjunction with the ongoing 2014 Harrier UG resource and reserve studies. The outcome will provide guidance/ support to future surface based diamond drilling at the Harrier UG Far South area.

9.6.4 May Reef

The May Reef Prospect is located in the north-eastern portion of EL3539, some 15 km north of the Fosterville Mine Lease. Several minor historic shafts (early 1900's) occur in the area including the May Reef shaft which is the namesake of the prospect. Shallow RAB drilling with follow up RC (8) drilling in the area through the unconsolidated gravel and clays to Ordovician turbidite bedrock identified gold and arsenic anomalism 100m west of the historical workings. The RC drilling in 1998 returned only one significant intersection (MR4: 10m @ 1.01 g/t Au from 42m incl. 2m @ 3.71 g/t Au). However, the context of the drill intercept is unclear and a review of the prospect is recommended.

9.6.5 Myrtle Creek

The Myrtle Creek prospect is located in the southern part of EL3539 on private land, 24km south of the Fosterville Mine Lease. The prospect is 4km northeast of, the 370Ma, Harcourt Batholith where rocks on the prospect comprise 440Ma Lower Ordovician Lancefieldian sediments, dominated by sandstone and quartzite, of the Castlemaine Supergroup. The sediments are tightly folded on an axis trending NNW, similar to that of other Bendigonian sediments east of the Whitelaw Fault. The sandstone-dominated sequence has been intruded by a granitic stock that measures 250 m by 200 m at surface, and by several quartz porphyry dykes up to 1.5 m wide, both of which may be related to the Upper Devonian Harcourt Granodiorite.

Gold was first discovered in the Myrtle Creek area in 1858 and sporadic mining for alluvial and quartz reef gold occurred up until the 1930's. Production from the goldfield is not well recorded, but James (2005) reported quartz reefs grading 1-2oz Au per ton. Modern exploration in the general Myrtle Creek area has occurred since 1974 by companies such as Noranda Australia (rock chip sampling, geological mapping, soil geochemistry (Au, Cu)), Ghana Gold (structural interpretation of aerial photography) and BHP (stream sediments and follow up soil surveys). Perseverance explored the area from the mid 1990's to 2006, completing regional stream sediment, rock chip and soil sampling, geological mapping and petrographic work on rock samples. Northgate explored the area between 2008 and 2009, undertaking additional surface sampling in the northern area of historical workings, but the results were disappointing with the overall tenor of gold-in-soil much lower than observed elsewhere on the prospect.

In 2009 Northgate drilled 10 diamond holes (totaling 1,695m) at Myrtle Creek to test a number of proposed mineralization settings including intrusion-related, fold-fault related, dyke-related and disseminated styles. Much of the drilling was centered about a 600m long by x 200m wide NW trending Au-Mo soil geochemical anomaly centered on the granite stock (Quartz Hill). The drilling, reported by Dean (2010), gained financial support of a drilling grant from the Rediscover Victoria Strategic Drilling Initiative.

Two of the holes returned significant intersections of gold mineralization are reported and interpreted to be from the NE trending New Amelia Mine Shear; Down hole widths of 10.9m @ 2.0g/t Au from 0.9m (incl. 6.0m @ 3.1 g/t Au from 4.0m) in hole MCD004 and 8.0m @ 1.9 g/t Au from 84.0m (incl. 2.0m @ 5.2 g/t Au from 88.0m) in hole MCD006.

Anomalous gold (7.61 g/t Au peak) and molybdenum (2,882 ppm) were encountered throughout much of the prospect, particularly in proximity to the granite. Visible gold was observed twice within sheeted quartz veins and there appears to be a strong intrusion-related Au-Mo-As correlation. A significant nugget-effect may be present given the presence of coarse gold and frequent highly anomalous As/Mo results without corresponding elevated Au.

The drilling at Myrtle Creek indicates that gold occurs in structurally controlled shears and is not disseminated widely through the wall rock. This fact caused Northgate to suspend exploration on the prospect. However, the drill intercepts on the New Amelia Shear remain untested along strike and down dip and this prospect is to be further reviewed by Newmarket Gold in the future.

9.6.6 Northern Area

The EL3539 Northern Area refers to the region north of the Goornong Township to the northern extents of the Exploration Licence. Previous exploration within the Northern Block was initially limited to BHP/Homestake exploration for northern extensions of the Fosterville and O' Dwyer's Faults. However, in the 1990's PSV carried out an extensive program of roadside geochemical sampling from which sporadic gold anomalies were defined and followed up by infill RAB and shallow RC drilling (an example of this being May Reef - best intersection of 2.0m @ 3.7 g/t Au).

The existing geochemical dataset, although widespread, does not presently cover the eastern portion of EL3539 where the area is now considered prospective for gold owing to its position in relation to the Redesdale Fault corridor.

The existence of the Redesdale Fault was first proposed in late 2009 by the Geological Survey of Victoria and is supported by the 2006 State seismic transect (which passes north of EL3539), geological mapping near Redesdale and interpretation of State and Northgate gravity data. The interpretation importantly defined a number of gravity highs within the Redesdale Fault corridor, corresponding with known areas of gold mineralization including the Fosterville and O' Dwyer's Fault Systems. This identified relationship is the basis for proposing future RAB drilling between the Goornong South and May Reef Prospects where similar, but less well-defined gravity anomalies are present.

9.6.7 Redesdale Fault Corridor

The current understanding of the underlying structural architecture in the Fosterville region provides opportunities for renewed greenfields exploration activity in relation to regional gold prospectivity. For instance there are a number of faults interpreted and mapped in the region that appear to have similar trends as the Fosterville Fault and abut the west side (hangingwall) of the Redesdale Fault. Examples within Newmarket Gold tenements include the Sugarloaf, O'Dwyer's and Drummartin Faults as well as other faults interpreted north of MIN5404 from Goornong South through the Northern Area to the May Reef gold occurrence.

Newmarket Gold will use the existing geophysical (airborne magnetics and radiometrics, gravity, ground IP), drilling and surface geochemical datasets to support renewed gold focused exploration activity in the Fosterville region.

9.6.8 Russell's Reef

The Russell's Reef Prospect is located within EL3539, approximately 2.4km south of the Fosterville Mine Lease. See Figure 9-1. The prospect is based on shallow historical shafts and pits spread over about a 250m north-south extent. Recorded historical production in the area totals 417 ounces from the 1897-1900 period of mining.

The area has been subjected to several lines of soil sampling, and several programs of shallow RC drilling (50 holes averaging 31m depth) undertaken over a protracted period from 1976 to 1989. Perseverance subsequently drilled nine diamond holes in 2006 to test for Fosterville style sulphide hosted gold mineralization. Three of the nine diamond holes returned drill intercepts averaging above 3.0 g/t Au.

These included:

RRD006:

- 4.0m @ 6.1 g/t Au from 48.0m (incl. 2.0m @ 9.4 g/t Au from 49.0m)

RRD005:

- 10.4m @ 2.2 g/t Au from 57.8m,
- (incl. 4.3m @ 2.9 g/t Au from 57.8 and 2.3m @ 3.1 g/t Au from 65.9m)

RRD007:

- 10.7m @ 3.1 g/t Au from 141.5m,
- (incl. 0.9m @ 7.5 g/t Au from 147.1m and1.4m @ 12.3 g/t Au from 150.8m)

Owing to Perseverance's drilling being heavily focused on exploration targets within the Fosterville Mine Lease, no follow up diamond drilling was undertaken at Russell's Reef.

9.6.9 Sugarloaf Range

The Sugarloaf Prospect area encompasses the entire length of the Sugarloaf Range, a ridge of steeply-dipping sandstone and quartzite located immediately west and south-west of the Fosterville Mine Lease. The prospect area is mostly within the Sugarloaf Nature Conservation Reserve.

A compilation and interpretation of available drilling and geochemical data in conjunction with interpretation of FGM' s airborne geophysical data (acquired in 2008) and consideration of Geoscience Victoria' s (GSV) Redesdale Fault Model indicates potential for Fosterville-style gold mineralization within the prospect area.

Exploration data in the area includes surface geochemistry, RC drilling, airborne magnetics and radiometrics and ground IP. However, it should be noted that historical (1989-1991) drilling of 36 RC holes (totaling 1,164m) in the area averages only 32m in depth and diamond drilling is absent.

Ground IP data, collected in 2010, maps resistive chargeability anomalies beneath the Sugarloaf Range and between the range and the Fosterville Fault. In addition to this an airborne radiometric K/Th ratio anomaly in the southern part of the prospect may represent an alteration halo proximal to faulting. The K/Th ratio anomaly also has a coincidental and similar trend to the Sugarloaf Fault IP chargeability anomaly. The chargeability anomaly could be caused by the presence of subsurface black shale stratigraphy and/or sulphides.

Newmarket Gold will review the prospect with the intent of undertaking ground exploration at the prospect in the future.

10 DRILLING

10.1 Pre-1992 Drilling

Section 9.1 described the work undertaken from the commencement of modern exploration in the 1970' s through to 1992.

10.2 1992-2001 Drilling

On acquiring the Fosterville Mine Lease in 1992, Perseverance (through a drilling contractor) started RC drilling for further oxide resources and reserves using a combination of cross over and face sampling hammers. These holes used the CN, CEL, CEN, DH and HAR prefixes.

In late 1994, while continuing to explore for oxide mineralization, Perseverance began to drill for sulphide mineralization on the Fosterville Fault potentially amenable to open cut mining. The 1997 Feasibility Study drilling was almost entirely RC with minor diamond drilling for metallurgical and geotechnical purposes and extended from 6000mN to 10700mN. Most of the drilling was completed by 1997 with minor infill drilling continuing to 1999. Holes from this program have the SP (sulphide project), CN, CEL (D), CEN (D), GT or HAR (D) prefixes, the 'D' denoting holes with a diamond tail (Table 10-1 and Table 10-2).

All the RC drill holes used face sample hammers. After 1996, if the sample was unable to be kept dry the hole was finished with an NQ2 diamond tail.

Open hole downhole surveys were completed on all drill holes at 30m intervals except for a small number of holes which collapsed before a survey instrument could be lowered down the hole. The vast majority of holes were drilled from the west towards the east, generally intersecting mineralization at 50-80°. Most sections include at least one hole drilled towards the west as a check on the geological interpretation.

The Fosterville Mine Surveyor used a Total Station Instrument to run a complete digital survey of the topography for any areas where drilling and later resource evaluation was planned to take place. Spot heights were measured at suitable intervals where easting, northing and RL are noted. Closer spaced measurements were taken around noticeable highs and lows in the topography. These spot heights were then triangulated using Minsurv software to construct a Digital Terrain Model (DTM). This DTM was used in all resource/reserve estimates at Fosterville. The spot heights were measured to an accuracy of ± 1.0cm at spacing of approximately 2m.

10.3 2001-2014 Drilling

The current drilling program largely on the Fosterville Fault Zone commenced in July 2001 and is ongoing. For the majority of this period, the surface drilling activities were conducted by Silver City Drilling Pty Ltd (drilling contractor), Macquarie Drilling Pty Ltd (drilling contractor) and Deepcore Pty Ltd (drilling contractor) since December 2009. All underground diamond drilling has and is being conducted by Deepcore Pty Ltd.

The SPD holes were drilled with RC pre-collars and NQ2 diamond tails. The diamond tails commenced at least 20 metres before the Fosterville Fault so that all mineralization was intersected by the diamond tail.

The RC pre-collars were generally 150m to 200m deep and the diamond drilling was double tube wireline drilling. In addition, navi or wedge drilling is undertaken from parent holes where holes depths are great, and since 2008 many of SPD prefixed holes were drilled using diamond drilling exclusively, HQ collars with NQ2 tails.

Collar locations are surveyed using the same technique as prior to 2001 (see Section 10.2 above).

The direction of the RC pre-collars was controlled to some degree by the use of a stabilizer rod, the relative size of the bit compared to the rods and by the weight on the hammer. Drill holes shallower than 70° tended to lift. Drill holes steeper than 75° tended to drop. With experience, deviation in the pre-collar was restricted to less than 1° in 10m. Navigational drilling was occasionally used to keep holes on target where the RC pre-collar deviated significantly. Down hole surveys were carried out using a single shot Eastman camera at 25m intervals in the pre-collars (every 50m inside the rods as the hole was drilled and the intervening 25m intervals open hole after the pre-collar was completed) and at 30m intervals in the diamond tails. As a check on the validity of the single shot surveys six holes were surveyed at 6m intervals using an EMS (electronic multi-shot) tool. Since 2010 holes greater than 130m have been surveyed at 6m utilizing the EMS tool on hole completion.

The drill hole traces are currently calculated using the 'semi tangent' de-surveying algorithm on 10m intervals in MineSight software. This method is suitable for deeper RC holes, which have more than two downhole surveys. The 'fit-spine' algorithm was previously used because it dealt well with RC holes that have only one or two surveys near the top of the hole and also because this algorithm was used historically at Fosterville.

The NQ2 diamond core has generally been drilled using either 6m core barrels for surface drill rigs or 3m core barrels for underground drill rigs. Core orientation is attempted with each 3m run predominantly utilizing an Ace Core Tool. Only 5% of structural measurements recorded in the Fosterville database are provided by orientation tools with 82% taken from an inferred reference plane (regional cleavage). 13% of structural measurements are un-oriented.

Sieved chips from the RC pre-collars were logged in two metre intervals for lithology, weathering, alteration, % quartz, color and recovery. The information was entered directly in the field into a hand held computer (IPAQ) and uploaded to the database. The uploading procedure has built in checks to prevent interval overlap, range checking etc. After uploading the entire log is printed for hand plotting and as a hard copy record. Since 2008 geological information has been entered into laptops running acQuire^{TM} Offline logging software, which supports increased validation options prior to uploading into the SQL Fosterville geology database.

The diamond core is transported to the core shed where the core is washed, oriented, geologically logged, recovery and RQD measured, marked up for sampling, digitally photographed, sampled and dispatched. Geotechnical logging occurs on an as needs basis, but is completed for each resource definition drill hole. The remaining core is stored on site either in the core farm behind the core shed or at a storage facility at the backfilled portion of the Falcon pit. The geological logging involves direct digital recording of observations on sediment grain size, lithology, planar and linear structural observations (as alpha, beta and gamma measurements), mineralization, alteration and quartz veining and identification of sample locations. When logging is complete it is uploaded into the database with the usual automated error checking and a list of samples printed as a cutting sheet. True dip and dip direction values for each collected structural measurement is calculated using a stored procedure in acQuire software. The logging used to be printed out for hand plotting and as a hard copy record, but since 2008 logged data has been verified through viewing of the data using MineSight 3D software.

The strategy for underground diamond drilling is to infill the exploration drilling intercepts (100m sections) to a notional 25m x 25m grid spacing (or tighter if required) prior to the mining of underground development. Underground diamond drill core samples used in the Phoenix and Harrier resource estimations were predominately NQ2 or LTK60 in size, depending on the drill rig used.

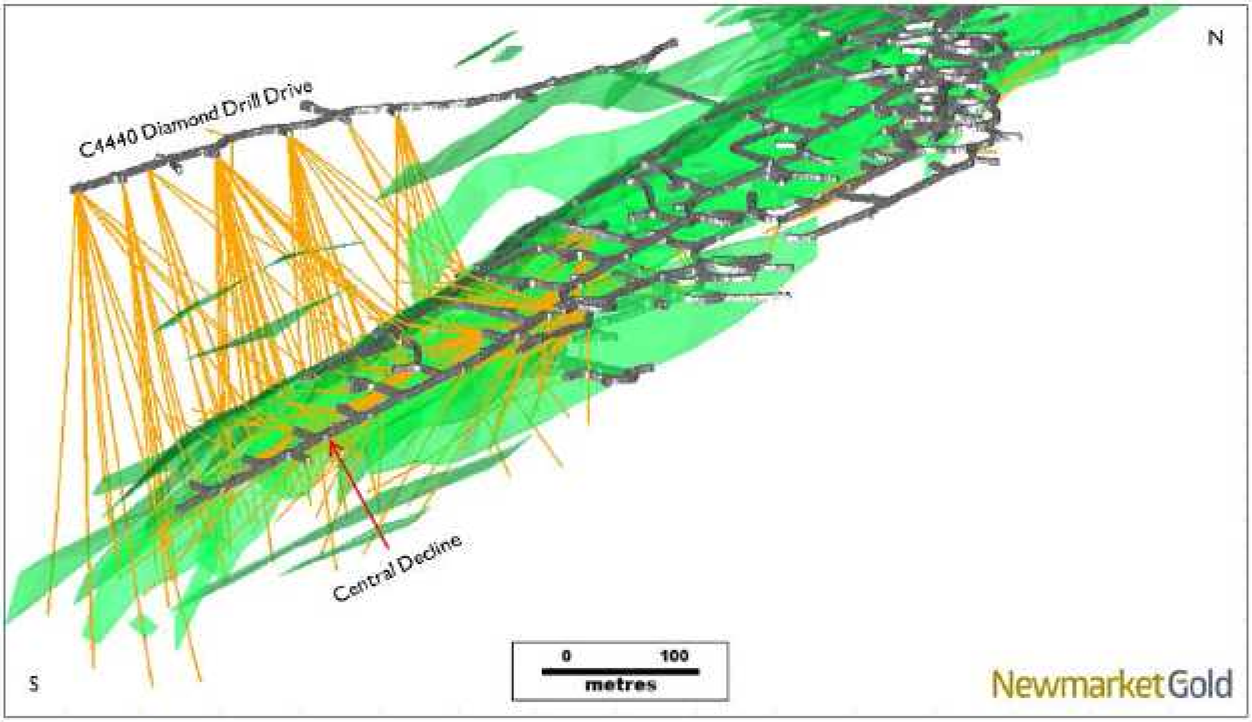

Figure 10-1 2015 Underground diamond drilling traces (orange) showing targeting of Lower Phoenix, Lower Phoenix Footwall from the C4440 Diamond Drill Drive and targeting of Eagle, East Dippers and Kestrel Structures from the Central Decline.

The nominal progression of drilling is from initial surface exploration drilling, through 100m by 50m and then 50m by 50m. Near surface mineralization is then further in-filled to 25m by 25m to allow pit design. Open pit grade control drilling consists of RC holes drilled 5m apart on either 10m or 12.5m sections to a maximum depth of 30m. However, for the Harrier pit cutback and the deepening of John' s pit, two 2.5m riffle split samples of 5m deep blast holes were used for grade control purposes. The open pit drilling, sampling and logging methods are the same as exploration RC drilling. Underground mineralization is in-filled to 25m by 25m or tighter if required by underground diamond holes.

Late 2014, it was identified that environments conducive to east dipping economic mineralization existed presenting the requirement for footwall drilling locations. The Central Decline development was accelerated to provide platforms to target and extend east dipping resources to the south. In 2015, roughly 39% of all drill meters were drilled into the Lower Phoenix and Lower Phoenix Footwall Zones, 27% into Eagle and East Dippers Zones, 18% into Phoenix and Kestrel Zones with the remaining 16% consisting of Exploration, Geotechnical and service hole drilling.

Strike drives are face sampled each round (~3m) and sludge hole sampled on 6m Northings in a ring pattern with holes selected by geologists after review of current geological information. The selection criteria for sludge sampling are based on either the need for providing diamond drill data support or the need for additional sampling in data poor zones. No face sampling or sludge hole sampling is used in reserve grade estimation however the information is considered for domain boundary placements.

Based on drilling results, geological interpretations are made in three dimensional surfaces to form a geological model. The geological model is utilized to interpret the mineralized zones, with geological solids subsequently generated from these interpretations. Further detailed discussion on this process is contained in Section 14 under each of the modeled areas.

Table 10-1 Drill hole prefixes for all drilling on the Fosterville Fault Corridor south of 10,000mN.

Hole Series		No. of Holes	Comments
BGL001	BGL106	33	1990- 2015 RC hydrological
CEL001	CEL124	96	1997 RC & AC open pit sulphide
CELD020	CELD106	26	1997-2003 Diamond tails from RC wet drilling
CELD051	CELD058	8	1996 Diamond metallurgical
CEM100	CEM105	6	1994 RC metallurgical
CEN001	CEN124	80	1997 RC for open pit sulphides
CEND019	CEND103	22	1997- 8 Diamond tails of RC
CEND110	CEND112	2	1997 Diamond Exploration
CEND038	CEND113	12	1996- 7 Diamond metallurgical
CN100	CN248	149	1994 RC exploration
CNM001	-	1	1995 RC metallurgical
DALD001	DALD020	21	2003-6 Daley's Hill diamond
DDH3*	DDH5*	3	1976 Daley's Hill diamond
DH001	DH238	193	1995-9 Daley's Hill RC
DHRB010	DHRB013	4	1997 Daley' s Hill RC
ELRC0001	ELRC0949	912	2005-7 Ellesmere pit RC (7500mN-8425mN)
FARC0001	FARC0825	825	2005 Falcon pit RC (8615mN- 8800mN)
FDD14A	FDD33	7	1990 Diamond (Brunswick)
FO002	FO379	235	1986- 90 RC (Bendigo Gold Associates)
FO400	FO487	56	1992-1994 RC (Perseverance)
FOS056	FOS214	3	1998-2000 RC & AC exploration
GT001	GT046	45	2004- 2015 Diamond geotechnical
H4805RAWPILOT	-	1	2014 Pilot hole for Harrier 4805 RAW
HAR003	HAR065	61	1997- 9 Harrington' s Hill RC
HARC001	HARC248	233	2006-11 RC (6350mN-7315mN)
HARD1	-	1	1996 Diamond PQ metallurgical
MB12	-	1	2009- 12 RC hydrological monitoring
SH003	SH016	14	2012 – 2015 Underground Services
SD001	SD039	43	2007-8 Diamond (7775mN-8675mN)
SP001	SP372	299	1994- 6 RC drilled down to 5100mRL
SPD001	SPD009C	9	1995 Diamond exploration
SPD010	SPD615D	691	2001-15 RC and diamond exploration
ST009	ST179	50	2003 RC & AC Sterilization

Hole Series		No. of Holes	Comments
SVH001	SVH009	9	2010 Underground Services
UD001	UD995	934	2006- 11 Underground diamond
UDE001	UDE114	125	2010- 15 Underground diamond exploration
UDH0001	UDH1550	1260	2011- 15 Underground diamond
	Total Holes	**6,470**	

Table 10-2 Drill hole prefixes for all drilling in the Robbin's Hill - O'Dwyer's Area.

Hole Series		No. of Holes	Comments
FAC001	FAC003	3	1993-2001 Farley's AC
FAR001	FAR011	10	1997 Farley's RC (face)
FARM001	-	1	1994 Farley's metallurgy RC (x-over)
FDD019	FDD040	12	1989-90 Robbin's Hill diamond HQ
FO303	FO309	6	1998 O'Dwyer's RC (face)
GH100	GH354	254	1993-96 Sharkey's RC (x-over) &
			diamond HQ (1) & NQ (1) & RAB (2)
GHM001	GHM002	2	1994 Sharkey's metallurgy RC (x-over)
MBOS01	MBOS07	7	2011 O'Dwyer's South RC hydrological monitoring
ODW001-134, 150-158 & 167		128	1999, (2005 ODW167) O'Dwyer's RC (face)
ODW135-149 & 159-166		23	1999 O'Dwyer's AC
ODW168	ODW206	39	2007 O'Dwyer's South RC (face)
ODW207	ODW228	22	2011 O'Dwyer's RC (17, face) & NQ2 (5)
ODWD001	ODWD003	3	1997 O’ Dwyer's diamond NQ
PBOS01	PBOS05	5	2012 O'Dwyer's South RC hydrological production
RD001	RD151	147	1994-98 Read's RC (83, face) and AC (64)
RDD146	-	1	1998 Read's diamond NQ
RH001	RH878	756	1987-96 Robbin's Hill and O'Dwyer's RC
RHD001	RHD207	204	1994, 2004 -07 Robbin's Hill RC &
			diamond NQ2 (47) & HQ (15)
RHD208	RHD241	34	2009-12 Robbin's Hill & Farley's-Sharkey's
			diamond NQ2 (25) & NQ3 (3) & RC (6, face)
RHM001	RHM004	4	1993 Robbin's Hill metallurgy RC (x-over)
ROB001	ROB012	11	1996 Robbin's Hill RAB
ROB013	ROB066	51	1998-99 Robbin's Hill RC (face) & AC (3)
SHA001	SHA033	25	1997 Sharkey's RC (face)
ST001	ST008	8	1993 Sterilization RC (x-over)
Total No. of Holes		**1,756**	

No drill holes are excluded from the database. However, drill holes that are of questionable quality (due to suspect collar co-ordinates, downhole surveys or sampling/analytical QAQC) are omitted from any resource calculation process. Such holes typically are in areas of historic mining and have no influence on the current Mineral Resource estimates.

10.4 QAQC of Drill Hole Surveys

Allwood (2003) details the results of downhole surveys repeated using both an Eastman camera and an Electronic Multi-Shot (EMS) tool. The EMS downhole surveys agreed with the single shot surveys to within 0.1 ° in dip and 2° in azimuth resulting in a total average variation of 0.4 m per 100 m downhole. The repeated Eastman surveys have an average variation of 0.6 ° in azimuth and 1.6 ° in dip, reflecting the precision of the Eastman camera survey tool. Comparing the drillhole traces plotted using the Eastman data with the EMS data shows that the variation in drillhole location due to survey method is considerably less than the variation in hole trace caused by the use of different drillhole de-surveying algorithms. However, in 2007 the use of EMS tools as a standard in preference to Eastman cameras was adopted across the various rigs operating at Fosterville, and in 2010 it became common practice to have survey data at 6m increments or less down each hole. The increased density of down-hole survey data has permitted ability to readily identify and remove suspect azimuth measurements.

Accuracy of downhole surveys are affected by very few factors at Fosterville. Drillholes can be affected when passing close to existing development due to steelwork (mesh, plates and cable bolts) associated with underground development; the effect is shown through elevated magnetic readings which allow the removal affected surveys. Over time the survey instruments accuracy degrades through usage. Tools are tested on a test bench of known dips and azimuths regularly to check and control any tool accuracy degradation.

11 SAMPLE PREPARATION, ANALYSES & SECURITY

11.1 Sampling Method and Approach

For the period 1987-1991 (Bendigo Gold Associates, then Brunswick), drilling used a cross over type hammer and significant down hole contamination is suspected for some holes below the water table where the rig was unable to produce dry samples. This drilling method is likely to have produced a relatively poor sample quality. During this time RC drill samples were collected at one metre intervals using 'spear' sampling. Although this sampling method is likely to have produced a relatively poor sample quality, the drilling was confined to oxide pits that have subsequently been mined and so is not relevant to the Resources and Reserves included in this report.

From the acquisition of the project by Perseverance in 1992 through to the present, all RC drilling through mineralization has been collected at one metre intervals and sampled as two metre composite samples. Prior to 1995, samples were collected using 'spear' sampling. Since 1995 all RC holes have been sampled using a riffle splitter split to either 12.5% or 6.25% depending on the drill hole diameter. After 1996, if the sample was unable to be kept dry the hole was finished with an NQ2 diamond tail. In the central area, spear samples comprise 16% of all mineralized samples and 28% of all mineralized RC samples. All RC holes were completely sampled.

As part of the 1997 Feasibility Study several of the FO prefixed holes (see Table 10-1) with long, high-grade intersections were twinned with RC holes drilled with a much bigger compressor and a face sample hammer resulting in dry samples. These twin holes demonstrated that there was significant downhole contamination in the FO holes (Perseverance, 1997). As a result, the FO holes were only used for estimating oxide resources and reserves where it is assumed that dry samples were recovered and downhole contamination was not an issue.

In the diamond drill core, all visible sulphide mineralization, quartz vein stockworks and LQ veins plus at least two metres of apparent waste either side is sampled. Samples are cut to geological boundaries and within a length range of 0.05m to 2m, with a preferred length of 1m. Infill diamond holes are full core sampled; the entire core sample is broken with a hammer in the tray and moved directly into the sample bag. All other core is halved along the plane of orientation using a diamond saw and the upper half of the core dispatched for analysis and the lower half returned to the core tray in its original orientation. The PQ core was sampled by cutting a sliver equivalent in volume to half NQ2 core from the top of the core. Recovery of diamond drill core is acceptable with >98% recorded for the drill holes incorporated into the Central Area Resource Models.

In underground sampling, an attempt is made to sample every round (3-4m nominal advance) in the ore drives where safe to do so. Sample intervals are chosen based on structure, mineralization and lithology, and are a minimum of 0.3m and a maximum of 1.5m in length. Mapping data that was collected at the same time as the samples is used to validate the sample results.

Figure 11-1 includes some 576 duplicate face sample pairs were collated include face sample duplicates taken on the Phoenix 4380m RL and the Phoenix 4280m RL. With outliers removed, the duplicates show a moderate correlation with an R^2 of 0.6402 (Figure 11-1). This study covered the underground face sampling method used throughout the mine since ore driving commenced in late 2006 to ore driving completed to the end of 2015. Face sampling data is used to refine resource domain boundaries. Sample grades from face sampling are not used in the resource estimation process.

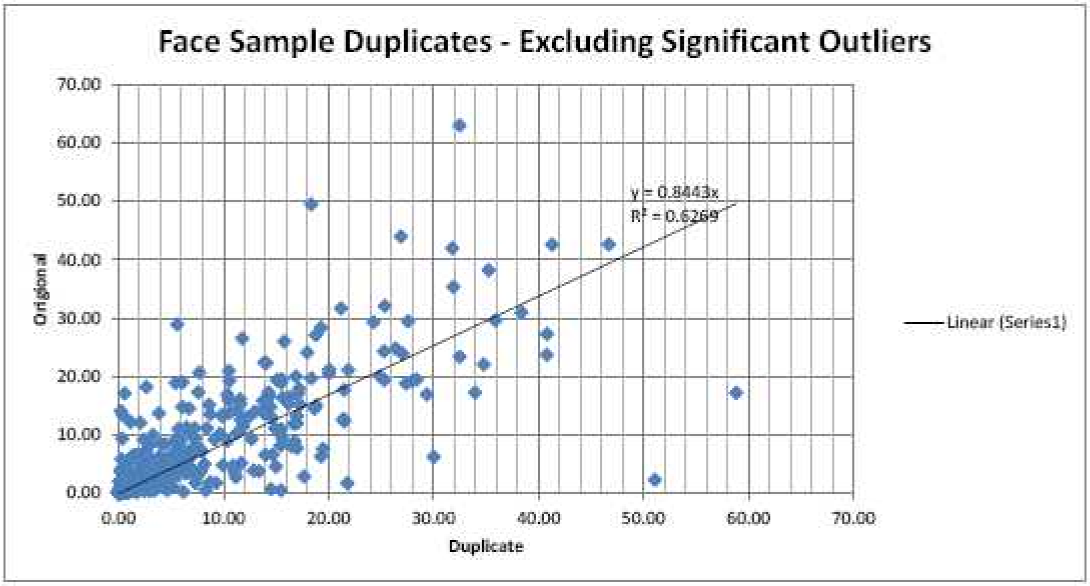

Figure 11-1 Underground face sample duplicate results.

Sludge holes are bored with 54mm diameter drill bits and sampled at 2m composite intervals giving a weight of between 2kg to 5 kg per sample. Cuttings are collected by a custom designed apparatus to maximize the catchment area to improve sample quantity/quality. Samples are inspected for quartz percentage, non-carbonate carbon content, sulphides present and lithology. Due to the poor quality of the samples, sludge samples are not used directly in resource estimations but may be used to define domain margins.

All remaining diamond drill core is stored on site within the fenced and gated core handling facility or within the mine compound on the backfilled Falcon Pit storage area. Assay sample pulps are also returned from the laboratory and stored at the core handling facility.

The RC samples from previous grade control programs were kept at an onsite depot for approximately three months after the receipt of final assay results. This allowed time for any re-sampling that may be necessary. The plastic sample bags photo-degrade rendering re-sampling impossible after six to 12 months and presenting an environmental hazard from windblown plastic, therefore the sample bags are disposed of as part of routine site rehabilitation works. Exploration RC pre-collar samples were collected in hessian sample bags since 2005 and similarly retained for a three month period at the drill sites. Hessian was chosen as it poses less of an environmental hazard and allows for mechanical rehabilitation of drill sites.

11.2 Elements

Table 11-1 Analysed elements by method and time period.

Element	Selection of samples
Au	All fresh samples by 40g Fire Assay to Dec 2004 and then by 25g Fire assay, all oxide samples by 25g aqua regia digest
As	All, since August 1995
S	For all Au values over 0.5g/t August 1995 to May 2001, then all samples
Sb	For all Au values over 0.5g/t August 1995 to May 2001, then all samples where stibnite is visibly discernable.
NCC (non-carbonate carbon)	For all Au values over 0.5g/t August 1995 to May 2001 From 2006, all sample intervals deemed as economic and 3m either side. . This is only for underground production hole
TGC (total graphitic carbon)	Selected samples between June 2002 and August 2003
TOEC (Total Organic and Elemental Carbon) - Equivalent to NCC	Selected RC pre-collar samples and most diamond samples since 1996
CO3	selected samples only
Cu, Pb, Zn, Ag, Fe, Mn, Mg, Bi, Ca, Cd, Ce, Co, Cr, K, Mo, Na, Nb, Ni, P, Sr, Ti, V, Y	All between May 2001 and Feb 2006, by ICP-AES, 5 gm HF mixed acid digest ICP3E, AMDEL
Ag, As, Bi, Ca, Cu, Fe, K, S, Sb	All since Feb 2006 by ICP-AES, modified triple acid digest, OSLS

The elements important to the metallurgical oxidation process are Au, S and Sb in decreasing importance. Arsenic is modeled for environmental reasons. NCC is of importance in the CIL stage as graphite is preg-robbing of gold in solution.

11.3 Description of Analytical Techniques

All of the gold analyses used in the sulphide resource model in the 2000 Sulphide Feasibility Study were fire assays of a 40g charge carried out by ALS at Bendigo, a commercial laboratory (non-accredited). The other elements were analyzed by a variety of techniques at a variety of laboratories. A full program of repeats, standards and inter-laboratory check sampling was conducted on the gold analyses. For the 2001 - 2004 NQ2 SPD diamond drilling campaign, gold analyses were determined by fire assay of a 40g charge by AMDEL in Adelaide, a commercial laboratory (ISO 9001 accredited). A 30 element suite including As, S and Sb was analyzed by ICP-AES from a separate 5g charge following HNO3/HF digestion. From November 2002 to August 2003 TGC (total graphitic carbon) was analyzed on a selective basis. A full program of repeats, standards and inter-laboratory check sampling was conducted on the gold analyses.
Since 2005, independent On Site Laboratory Services (OSLS), a commercial laboratory based in Bendigo, has been the primary provider of analytical services to the operation. The OSLS Bendigo laboratory gained ISO 9001 accreditation in October 2008 with registration ISO9001:2008 (CERT-C33510).

OSLS use a combined crusher and mill to pulverize the entire sample to a nominal 90% passing 75µm. A 25g sub-sample is analyzed for gold by fire assay with an AAS finish. A 0.5g sub-sample of the pulp is digested in a HNO3/HCl digest and then analyzed for Ag, As, Bi, Ca, Cu, Fe, K, Sb and S by ICP-AES. A full program of repeats, standards and inter-laboratory check sampling was conducted on the gold analyses.

An audit of the OSLS facility was completed for Perseverance by an external consultant during 2007 (Stewart, 2007). This audit found that OSL' s procedures were adequate and presented no major risk to the resource estimate. There were areas for improvement identified with the following corrective actions taken during the second half of 2007:

- Temperature variation within the drying oven is now being measured and recorded.
- Sizing analysis for all pulps is now being conducted and recorded.
- Calibration of scales is now being recorded and documented.
- Further improvements also included AAS electronic data capture in 2011
- Fosterville staff have formal monthly laboratory meetings to discuss performance

Work undertaken by employees of Fosterville is limited to core logging and the mark-up, cutting and bagging of samples. All other sample preparation and analysis was conducted off-site at the commercial laboratories.

Since April 2015, Gekko Analytical Laboratories (GAL) have been contracted to provide analytical services for diamond core and underground face samples. Analytical techniques include fire assay for gold, titration and atomic absorption spectrometry for Antimony, combustion analysis and Infrared detection for both sulphur and Non-organic Carbon. Gekko Analytical Laboratories gained National Association of Testing Authorities, Australia accreditation (NATA) in October 2015 with accreditation number, 19561.

All samples are dried at approximately 105 degrees. GAL use a Jaw crusher to crush the sample material to 8mm. The sample is then placed within a Boyd crusher and rotary splitter combination to enable further crushing to 3mm and optional splitting of the sample if it weighs in excess 3kg. Pulverization takes place with up to 3kg of sample to achieve 95% passing 75um. Sizing is reported with Au assays at 1:20 frequency. Approximately 120g of pulverized sample is scooped into a wire and cardboard pulp packet. Two pulp packets are created as a lab duplicate at a frequency of 1:10. A 25g scoop of sample is taken from the pulp packet and smelted with 180g flux. A 10g scoop from the pulp is re-fired for comparison if the initial grade was determined at >50g/t. Antimony is analyzed by aqua regia digest and AAS finish. If the result is over 1%, the sample is then analyzed by an acid digestion and titration. Total sulphur is analyzed using combustion analysis followed by Infrared detection. Non Carbonate carbon is analyzed by weak acid digest and combustion analysis followed by Infrared detection.

A laboratory audit in June 2015 was conducted by FGM personnel at the GAL facility situated within Ballarat. Its primary focus was to assess the preparation and handling of FGM' s sample material through the GAL laboratory and to observe the data and material handling process. No major risks were observed.

11.4 QAQC of Assays

Fosterville uses independent assay laboratories, which provide assay data in digital form. Since July 2007 OSLS has been the main assay laboratory used for assaying drill and grab samples. Since April 2015 some analytical assay work has been completed with GAL and has resulted in a successful trial from April to August in the same year, culminating in an agreement to send ~20% of total drilled diamond core samples and all production face samples, which is in place to date.

The laboratory digital data is imported directly into the database with a variety of automatic quality control checks preventing sample number mismatches, sample interval overlap, etc.

An assay QAQC program is used at Fosterville and has been improved and developed over the years. The system comprises four main strands with the reliance on standards, duplicates, repeats and blanks samples. Each strand is summarized below.

Standards

Drilling programs up to the end of 2007 included the use of four gold mineralized standards provided by Gannet Holdings Pty Ltd (ST148, ST109/0285, ST73/7192 and ST43/7194) and one standard prepared from approximately 500kg of Fosterville sulphide mineralization from previous RC drilling (AA). Over time the use of gold mineralized standards from Gannet Holdings Pty Ltd has diminished with only two (ST161, ST148) currently in use. During 2015, OSLS reported the following laboratory standards in their assay report files in Table 11-2.

Table 11-2 OSLS Laboratory Standards, g/t Au

STANDARDID	Mean	Mean- 3SD	Mean+3SD
ST345	55	40	70
ST588	1.60	1.45	1.75
ST37/6373	1.69	1.48	1.90
ST398	4.87	4.24	5.50
ST507	4.94	4.28	5.60
ST484	7.52	6.59	8.45

Similarly, GAL during 2015 reported the following laboratory standards in their assay report files in Table 11-3.

Table 11-3 GAL Laboratory Standards, g/t Au

STANDARDID	Mean	Mean- 3SD	Mean+3SD
ST528	0.51	0.42	0.60
ST431	1.54	1.36	1.72
ST383	7.24	6.43	8.05
ST484	7.52	6.62	8.42
ST448	13.29	11.52	15.06
ST335	13.65	11.79	15.51
GLC911-3	1232	1064	1400

At the laboratories, OSLS and GAL insert the certified standards into Fosterville assay jobs. These fall close to the expected values with anything outside of the expected ranges is investigated as per laboratory and company procedures.

Between 2008 and 2014 a further fifteen gold mineralized standards from Geostats Pty Ltd were adopted for use at Fosterville, with 10 in current use. Of the active standards a selection of four standards are "in use" at any one time. The "in use"standards are inserted into assay jobs at the rate of one approximately every 40 samples, which provides enough data for any given “in use” standard for assessment of laboratory performance on a monthly basis.

During 2015 the use of Fosterville Gold Mine standards continued. The gold mineralized standards have a wide range of gold grades extending from less than 0.3 g/t Au to about three times the average ore grades expected at Fosterville.

Gold mineralized standards sent to OSLS (since 2005) have returned gold assay results that on average fall very close to the expected values. All the values falling outside the expected ranges were investigated and where appropriate, batches are either re-assayed from stored pulps or re-sampled from remaining drill core. A similar protocol of re-testing the standards is employed at GAL.

The use of “pigeon pairs” is a key part to managing the “expected” value of any given standard in both the GAL and OSLS laboratories. The pigeon pairs form part of the current standard reference materials. The pigeon pairs are a pair of standards that have mean reference values relatively close to each other such that they share a part of their respective 3SD sample range around the mean expected value as an overlap with the other standard used in the pair. This is to create an element of uncertainty where the standard may be in use with a laboratory for a period exceeding 1 - 2 years. Only one member of the pair is used at a given laboratory at one time.

Duplicate Samples

Field and laboratory pulp duplicates are used to assess sample representivity at the field and laboratory pulverization stages.

Assays of field duplicates test sample representivity of the Fosterville half-core sampling process. During the 2013 - 2015 period, field duplicate data had a correlation coefficient value (R^2) value of 0.96, with no apparent bias as represented within Figure 11-2. During 2015, the field duplicate data was a combination of assay information from OSLS and GAL.

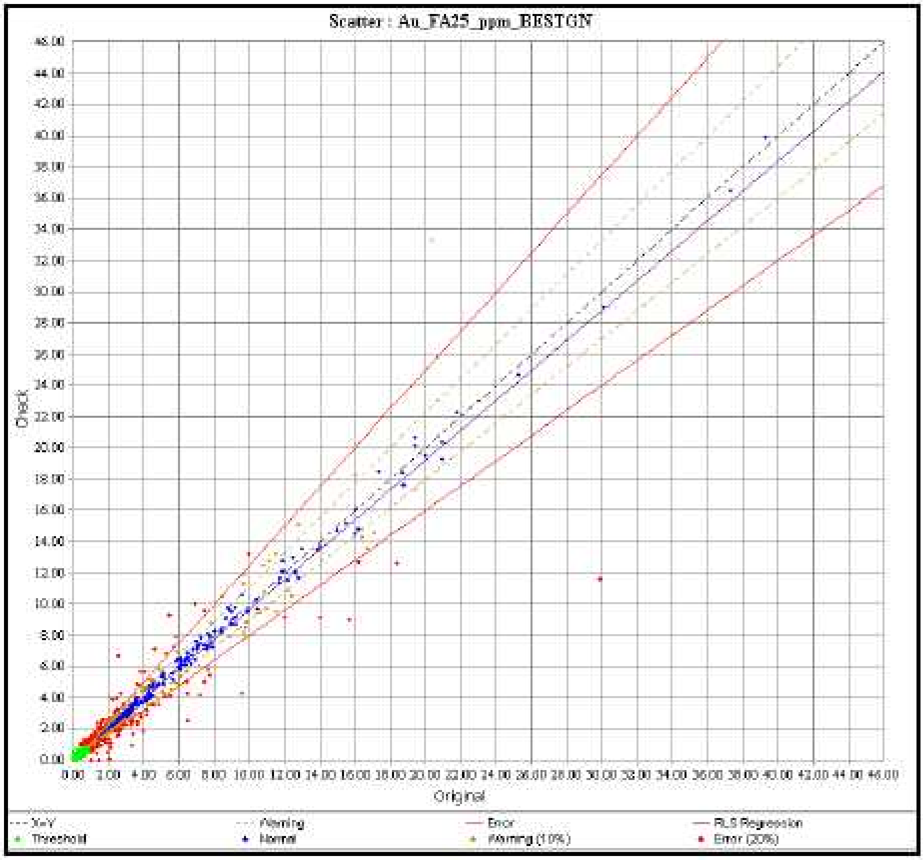

Figure 11-2 Field Duplicate Data to 31st December 2015

Laboratory pulp duplicates are used to test for sample homogeneity and errors of analysis. The laboratory duplicates are conducted at a rate of 1 in approximately 10 samples, and are assessed against expected performance levels of the analyses method. Any sample found to be outside the expected performance levels are investigated and where appropriate, batches are re-assayed from stored pulps or re-sampled from remaining drill core.

Laboratory duplicates of gold are highly repeatable with R^2 correlation coefficients of 0.99 for both AMDEL and OSLS data. There was also a program of inter-laboratory check assays undertaken in 2002 comparing the AMDEL results to two other commercial laboratories – Aminya Laboratories Pty Ltd (Aminya) and Genalysis Laboratory Services (Genalysis). The two batches (147 samples) sent to Aminya returned an average of 9% higher with an R^2 correlation coefficient of 0.993. The Genalysis results were 2% lower with an R^2 correlation coefficient of 0.996. The inter-laboratory check samples range in grade from below detection (<0.01 g/t Au) to 45 g/t Au. All the inter-laboratory check data is presented in Allwood (2003).

During 2013 OSLS' s performance was compared to another commercial laboratory, GAL in Ballarat. A total of 245 samples were submitted to GAL for inter-laboratory check assays with OSLS results returning 2% lower, with a correlation coefficient (R^2) of 0.988.

During 2015, laboratory duplicate data was a combination of both OSLS and GAL and over the last 3 years the OSLS data has correlated very strongly with an R^2 correlation coefficient of 0.98, similarly the GAL data during 2015 correlated very well with an R^2 correlation coefficient of 0.94.

Repeat samples

Laboratory repeat samples (subsamples of the original pulp sample) are used to test for errors in laboratory analysis by testing the sample pulp in a different batch of fire assay samples. The repeat samples are randomly chosen as well as selectively, where original batch fire sample results are abnormally high or low.

Repeat samples are assessed against an expected performance level and are investigated, where appropriate, batches are re-assayed from stored pulps or re-sampled from remaining drill core.

Blanks

Field blanks were historically not used because there is a sharp visual grade contrast between mineralization and waste which provides a natural blank. However, in 2009 the use of field blanks was adopted to assess quality control of the sample preparation; i.e. to test for contamination from one job to the next and also from sample to sample within the job.

Two field blank samples are inserted into each diamond drill hole sample batch submitted to OSLS. One field blank sample is inserted at the beginning of the job, with a second inserted between mineralized samples.

Since August 2012 laboratory blank samples have been imported and assessed as part of the FGM QAQC process for drill core.

Since June 2014 interstitial blanks have been routinely inserted within visible gold zones of holes containing potential or observed visible gold.

11.5 Sample and Data Security

11.5.1 Sample Security

Sample security information has not been recorded over the history of the project. However, to the best of the Authors' knowledge, the methods of sample storage and transport have remained largely unchanged throughout the life of the project.

Samples are bagged and numbered either on site at the drill rig or at the onsite core handling facility.

Samples sent to laboratories outside Bendigo were sealed in bags in lots of about ten and sent using commercial freight companies with tracking systems to the relevant laboratories. On arrival at the laboratory, the list of samples sent is matched to the actual samples received and confirmation is sent by either fax or email.

Analytical laboratories have operated in Bendigo during the periods 1992 – 2000 and 2005 to present. During these periods individual samples from the drill rig or core shed have been placed in a container within the mine security gate and collected daily by laboratory staff. Again, on arrival at the laboratory, the list of samples sent is matched to the actual samples received and confirmation is sent by either fax or email.

Work undertaken by employees of Fosterville was limited to core logging and the mark-up, cutting and bagging of samples. All other sample preparation and analysis was conducted off-site at commercial laboratories.

11.5.2 Data Security

Data security is ensured through the use of an 'acQuire/SQL Server' database of all company exploration drilling information. This database includes all assays, geological and geotechnical information. As well as data interrogation, the database allows automated error checking as new data is entered. The database is backed up to tape incrementally daily and fully weekly and annually.

Access to the database is controlled by user login permissions and the acQuire software.

11.6 Adequacy of Procedures

It is the opinion of the Authors that the sample preparation, security and analytical procedures are adequate and have been appropriately applied over the life of the project to ensure that the data is representative and of high quality.

12 DATAVERIFICATION

12.1 Database Validation

The drilling carried out by previous owners at Fosterville routinely included quality assurance and quality control checks. The nature of these checks evolved through time and these are described below. In addition, sampling QAQC consultants SMP Consultants reviewed the sampling, analytical and data storage procedures used in drilling programs to May 2002 (Crase, 2002). Data systems reviews of the exploration database were also undertaken by IO Digital Systems in 2004 and 2006 (Kelemen, 2004; McConville, 2006).

The database includes numerous automated data validation methods. The database structure and the use of primary key fields prevent certain types of invalid data (e.g. overlapping sample intervals) from being stored in the database. Also, numerous checks are performed on the data when it is imported (e.g. assay QAQC performance gates, variation in downhole surveys from previous survey).

Prior to 2000, the geological data was entered directly into the database by hand from the original hardcopy geological log with a manual validation system. From 2001 until 2008, all geological data was uploaded directly from IPAQ hand held logging devices into the database with similar automatic checks as used for the assays. Immediately after the IPAQ is uploaded a hard copy of the geological log is printed to provide an extra back up of the data. Since 2008 geological information has been entered into laptops running acQuireTM offline logging software. This software supports an increased range of logging validation that prompts the user while logging and also prior to uploading of the logged data into the Fosterville Geological SQL database.

The downhole drilling survey data, between 2001 and 2010, was the only data hand entered into the Fosterville geology database. Allwood (2003) reports a program conducted in 2002 where approximately 10% of the SPD holes were randomly selected for checking the database against the original survey shots. This check found several errors so it was decided to check the entire downhole survey database against the original surveys shots. All errors found were corrected. Diamond drill hole (underground holes are prefixed by UD, UDE and UDH) traces are visually checked in MineSight software against the design trace, as soon as the downhole surveys are entered into the database.

12.2 Data Verification

In addition to the quality control and data verification procedures discussed in detail above, the Qualified Persons preparing the Mineral Resource estimates have further validated the data upon extraction from the database prior to resource interpolation. This verification used MineSight drill views as the primary tool to identify data problems. This allowed the omission of holes if they were of questionable quality, for example due to low quality sample techniques or incomplete assaying. When coupled with the more mechanical check processes ensuring high quality data is entering the database in the first place, these checks were effective in allowing the Qualified Persons to be confident that the data was geologically coherent and of appropriate quality and adequate for use in resource estimations and reserve studies.

13 MINERAL PROCESSING AND METALLURGICALTESTING

The following section details the metallurgical test work conducted on a range of Fosterville ores, with particular focus on the test work that contributed to the Fosterville Bankable Feasibility Study in 2003 (Perseverance, 2003) Multiple batch flotation test work campaigns and two pilot flotation test work campaigns were completed for Perseverance by Metallurgy International (MI), Amdel Limited (Amdel), and Ammtec. The biological oxidation technologies offered by both Bactech (Australia) Ltd (Bactech) and Goldfields Limited (GFL) were extensively tested and the latter technology was selected.

Samples used for test work were selected by Fosterville personnel in conjunction with Mr. David Foster of MI. Effort was made to ensure, where possible, the samples selected for test work were representative of the respective mineralized zones that make up the resource. The major zones of mineralization Phoenix, Falcon, Ellesmere and Harrier were all subjected to test work in 2003 as either major composites or variability samples. Additional samples from Harrier Base, Osprey and Raptor were collected in 2008, and subjected to further flotation test work.

The location of the metallurgical test work samples within the various mineralized structures is provided on the long projections presented in Figure 13-1 and Figure 13-2.

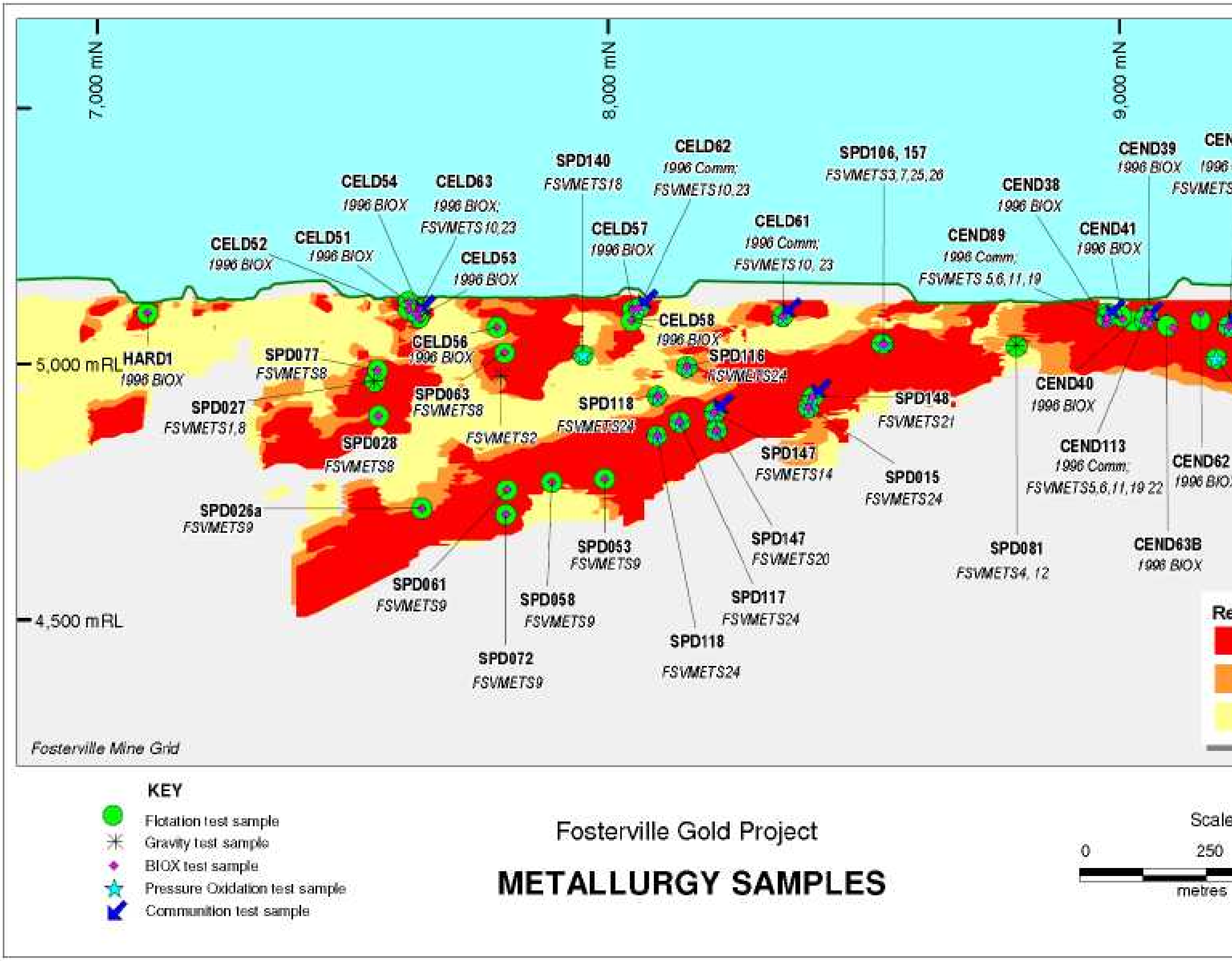

Figure 13-1 Longitudinal projection showing location of the 1996-2003 metallurgical testwork.

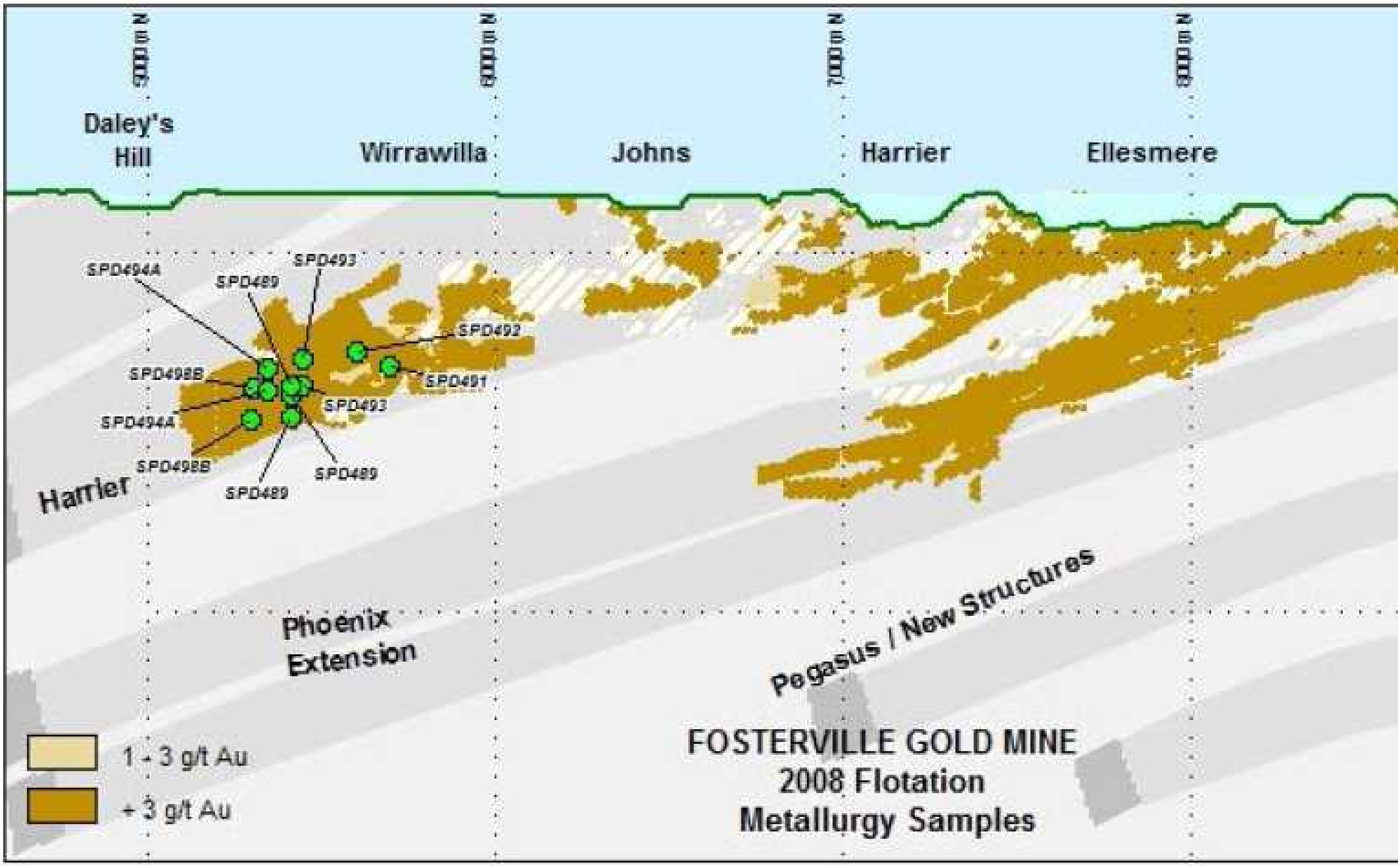

Figure 13-2 Longitudinal projection showing location of the metallurgical testwork conducted in 2008.

13.1 Results

Flotation

A program of batch flotation test work (December 2002 to February 2003) was conducted at Ammtec on a range of composites from each of the Phoenix, Falcon and Ellesmere Mineralized Zones and on fresh and weathered (clay) ore types. In January 2009, additional batch floatation test-work was carried out on composite samples from fresh ore types from the Harrier Base, Raptor and Osprey Zones. Results are reported in Table 13-1. Under laboratory conditions, the flotation test-work carried out exceeded gold recoveries of 96.5% and Sulphide recoveries of 92%.

Since the commissioning of the plant in 2004, all flotation models have been based on actual plant performances. Data pertinent to flotation performance is discussed in Section 17: Recovery.

Several newly discovered geological structures at depth, such as Eagle, East Dipping and LPFW Faults, have gold in the form of coarse visible gold that frequently occurs with sulphide mineralization. In 2015, a series of plant trials and mineralogy surveys indicated that the visible gold is being recovered in the flotation concentrates (primarily Flash flotation concentrate) and is recoverable from this concentrate by gravity methods. Newmarket Gold has now committed to installing a gravity circuit in the first half of 2016, (Knelson concentrator and Gemeni tables) treating the concentrate regrind circulating load.

BIOX

A number of concentrate samples were generated over the same period (December 2002 to February 2003) for BIOX® and cyanide leach testing. BIOX® variability test-work results are reported in Table 13-2.

Based on the BIOX® variability test-work, the average sulphide oxidation anticipated was 96.6% . With approximately 80% of the gold present associated with the arsenopyrite. The oxidation of sulphides is acid-generating and the use of limestone was determined to be adequate to maintain the pH between a range of 0.8 – 1.2. The laboratory test-work indicated that up to 140 kg of lime per tonne of BIOX® feed would be required to sustain the optimum pH conditions for the bacterial colonies.

The BIOX® bacteria are sensitive to chloride levels in the water, and management of BIOX® feed dilution water quality to <1000ppm Cl^- is critical for the health of the BIOX® circuit. Likewise, cyanide and thiocyanate species are also toxic materials to the bacteria, hence the Flotation and Neutralization waters, plus CIL decant liquors are managed separately at the Fosterville operations to eliminate any processing risks.

The BIOX® bacteria are also sensitive to elevated concentrations of antinomy >10%. Fosterville operations have control processes established to ensure antinomy levels are effectively managed in feed through material blending practices as well as supporting BIOX® monitoring controls.

Since the commissioning of the plant in 2004, all BIOX® performance models have been based on actual plant performances. Data pertinent to BIOX® performance is discussed in Section 17: Recovery.

Leaching

Leaching variability test-work has been conducted on the oxidized BIOX product. Results are reported in Table 13-3 and exceed approximately 90% gold recovery.

Since commissioning in 2004, non-carbonate Carbon, or NCC, found in the Fosterville Orebodies in the form of bituminous coal has presented a significant gold recovery loss through the leaching facility through 'preg-robbing'. The discovery of the significance of heat on leach recoveries triggered extensive test-work and a detailed design into the application of 'Heated Leach' technology commenced in September 2008 with commissioning finalized in Q3 2009. The advent of the Heated Leach circuit (Figure 16-2) has proven itself to be a significant contributor to the overall plant performance (Table 17-1).

Since commissioning the conventional leach plant in 2004, and the Heated Leach plant in 2009, all leaching performance models have been based on actual plant performances. Data pertinent to leaching performance is discussed in Section 17: Recovery.

In the opinion of the authors, all deleterious elements are effectively managed and it is considered that their presence does not have a significant impact on economic extraction. No identified processing factors have a significant impact on economic extraction.

Table 13-1 Floatation test results.

Ammtec Test Number	Fosterville Sample ID	Calculated Head Grade					Cumulative Laboratory Float Time	Concentrate									
								Mass Pull	Cumulative Concentrate Assays					Cumulative Recovery to Concen			
		Au g/t	S %	As g/t	CTOT %	CORG %	Minutes	%	Au g/t	S %	As %	CTOT %	CORG %	Au %	S %	As %	Cor
Phoenix Ore																	
RG4641A	FSV3 1D	5.53	2	5000	0.82	0.08	25	7.84	69.73	25	6.1	0.57	0.36	98.77	98.43	96.39	37.
RG4674	FSVMETS9	6.8	2.06	6900	1.04	0.11	45	11.59	57.8	17.3	5.9	0.84	0.47	98.49	97.42	98.66	50.
RG4790	FSVMETS14	2.96	0.95	4900	1.05	0.08	45	9.67	29.6	9.6	5	1.05	0.41	96.49	97.62	97.79	46.
RG4681	FSVMETS20	5.26	2.57	5500	1.07	0.13	45	13.06	39.71	19.4	4.2	0.72	0.43	98.6	98.3	98.4	41.
RG4791	FSVMETS21	5.25	1.57	7400	1.2	0.1	45	10.81	47.7	13.9	6.8	0.89	0.4	98.13	96.01	98.68	44.
Falcon Ores																	
RG4672	FSVMETS5	12.81	2.76	10400	0.95	0.19	45	13.13	95.16	20.7	7.8	1.12	0.9	97.52	98.43	97.83	62.
RG4686 *	FSVMETS6	11.3	2.42	9800	0.17	0.14	45	12.9	84.59	18.8	7.6	1.33	1.07	97.84	100	100	100
RG4685	FSVMETS11	10.33	2.27	9700	0.14	0.1	45	12.42	81.7	18.1	7.9	1.14	0.77	98.21	99.04	100	100
RG4782	FSVMETS22	10.81	2.31	5700	0.92	0.14	45	11.13	94.17	20.6	4.8	1.07	0.78	96.92	99.04	92.78	62.
Ellesmere Ores																	
RG4673	FSVMETS10	2.88	1.21	4100	0.61	0.12	45	7.56	34.36	15.7	5.2	1.2	0.9	88.31	98.09	95.05	54.
RG4783	FSVMETS23	3.75	1.34	3400	0.65	0.08	45	8.36	44	15.8	3.8	0.86	0.45	98.16	98.29	95.09	44.
RG4675	FSVMETS8	4.92	2.02	5500	1.09	0.12	45	10.21	47.42	19.3	5.3	1.13	0.73	98.36	97.77	98.05	62.
Osprey Ore																	
GS3912	Fresh	8.02	1.95				20							99.7	99.6		
Raptor																	
GS3913	Fresh	2.04	0.84				20							96.8	92.3		
Harrier Base																	
GS3914	Fresh	5.38	1.74				20							99.2	95.9		

Table 13-2 Summary of BIOX® variability test work.

Sample ID	Orebody/Ore Type	Sample	Mass Gain %	Reagent Consumption		Sulphide Oxidation %	Arsenic Dissolution %
				Lime kg/t BIOX® Feed	Net Acid kg/t BIOX® Feed		
FSVMETS 5	Clayey Falcon Ore	Sample A	34.6	181	-238	95.0	93.1
		Sample B	19.4	201	-302	91.5	94.3
FSVMETS 6	Clayey Falcon Ore	Sample A	2.8	137	-196	98.5	95.9
		Sample B	1.1	166	-253	96.9	95.0
FSVMETS 7	Fresh Phoenix Ore	Sample A	5.9	137	-200	96.8	93.6
		Sample B	2.8	166	-246	97.6	95.1
FSVMETS 8	Fresh Ellesmere Ore	Sample A	18.5	163	-244	99.2	80.5
		Sample B	8.9	129	-179	99.9	89.3
FSVMETS 9	Fresh Phoenix Ore	Sample A	23.4	163	-241	98.7	40.5
		Sample B	12.3	150	-211	99.3	46.8
FSVMETS 10	Clayey Ellesmere Ore	Sample A	2.6	110	-157	99.9	95.1
		Sample B	2.9	118	-171	99.1	93.6
FSVMETS 20	Fresh Phoenix Ore	Sample A	2.1	104	-131	99.2	44.1
		Sample B	32.9	144	-214	98.9	29.1
FSVMETS 21	Fresh Phoenix Ore	Sample A	-2.4	85	-112	71.1	45.8
		Sample B	6.6	141	-197	99.9	82.3
FSVMETS 24	Fresh Phoenix Ore	Sample A	2.6	118	-161	96.8	53.4
		Sample B	-1.1	111	-152	98.9	59.8
Average			**9.8**	**140**	**-200**	**96.6**	**73.7**

Table 13-3 Summary of cyanide leach of BIOX® variability residues.

Sample ID	Orebody/Ore Type	Testwork Method	Reagent Consumption Lime ® kg/t BIOX Feed	Reagent Consumption Net Acid ® kg/t BIOX Feed	Residual Cyanide CN FREE ppm	Gold Extraction %
FSVMETS 5	Clayey Falcon Ore	As Received	0.1	11.3	4,119	3.0
		BIOX® Residue A	2.3	12.5	2,393	92.9
		BIOX® Residue B	1.8	12.2	2,930	93.5
FSVMETS 6	Clayey Falcon Ore	As Received	0.4	13.0	3,684	8.3
		BIOX® Residue A	2.3	13.8	2,582	94.2
		BIOX® Residue B	1.3	12.4	2,437	92.5
FSVMETS 7	Fresh Phoenix Ore	As Received	0.2	9.9	4,467	0.7
		BIOX® Residue A	2.2	14.8	2,379	94.1
		BIOX® Residue B	1.3	12.9	2,698	90.3
FSVMETS 8	Fresh Ellesmere Ore	As Received	0.0	27.3	696	4.6
		BIOX® Residue A	5.2	17.4	1,769	91.7
		BIOX® Residue B	4.2	17.5	2,292	93.1
FSVMETS 9	Fresh Phoenix Ore	As Received	0.0	28.5	348	2.1
		BIOX® Residue A	23.3	19.0	1,566	95.8
		BIOX® Residue B	9.9	23.3	406	85.2
	Repeat	BIOX® Residue B	12.6	17.8	3,553	94.7
FSVMETS 10	Clayey Ellesmere Ore	As Received	0.1	12.7	3,742	37.7
		BIOX® Residue A	2.6	16.0	2,350	95.0
		BIOX® Residue B	1.5	13.4	2,495	93.2
FSVMETS 20	Fresh Phoenix Ore	As Received	0	27.8	493	4.6
		BIOX® Residue A	27.2	25.5	1,102	88.5
		BIOX® Residue B	53.2	28.0	1,944	91.7
FSVMETS 21	Fresh Phoenix Ore	As Received	0	25.9	1,015	1.7

Sample ID	Orebody/Ore Type	Testwork Method	Reagent Consumption		Residual Cyanide CN FREE ppm	Gold Extraction %
			Lime ® kg/t BIOX Feed	Net Acid ® kg/t BIOX Feed		
		BIOX® Residue A	6.4	13.8	3,626	95.3
		BIOX® Residue B	6. 9	15.4	3,481	94.0
FSVMETS 24	Fresh Phoenix Ore	As Received	0.02	21.5	841	8.6
		BIOX® Residue A	13.9	23.7	1,653	89.3
		BIOX® Residue B	10.9	20.2	2,089	89.7
Average		**BIOX® Only**	**9.9**	**17.0**	**2,408**	**92.7**

14 MINERAL RESOURCE ESTIMATES

The Mineral Resources reported are broken down into areas contained within the mine lease MIN5404 (Section 4). Mineral Resource Areas of Central, Southern, Harrier and Robbins Hill (Table 14-1) are defined resource areas which were established at different times in the projects history. The Central and Robbins Hill Areas contain multiple mineral resource models primarily for reasons of data handling. Details on mineral resource block model extents can be seen in Figure 14-1.>

CIL Residue Mineral Resources are distinguished from insitu Mineral Resources in Table 14-1 on the basis of differing recovery assumptions.

The current Mineral Resource estimate for the FGM is presented below in Table 14-1.

Table 14-1 Mineral Resources (Inclusive of Mineral Reserve) for FGM as at December 31, 2015.

Mineral Resources (Inclusive of Mineral Reserves) - Fosterville as at December 2015										
Classification		Measured			Indicated			Inferred		
		Tonnes (kt)	Grade (g/t Au)	Insitu Gold (kOz)	Tonnes (kt)	Grade (g/t Au)	Insitu Gold (kOz)	Tonnes (kt)	Grade (g/t Au)	Insitu Gold (kOz)
Fosterville Fault Zone Sulphide Resources										
Central Area	Upper	1,463	2.47	116	808	2.69	70	24	1.45	1
	Lower	315	8.29	84	5,188	6.65	1,109	1,488	5.58	267
Southern Area	Upper	21	3.32	2	463	2.44	36	537	2.29	40
	Lower	0	0.00	0	0	0.00	0	320	5.59	57
Harrier Area	Upper	0	0.00	0	0	0.00	0	0	0.00	0
	Lower	17	5.03	3	2,720	5.65	494	1,266	5.64	230
Robbin's Hill Area Sulphide Resources										
Combined	Upper	0	0.00	0	1,787	1.77	102	976	1.51	47
	Lower	0	0.00	0	114	3.81	14	59	3.38	6
Sulphide Upper		1,484	2.48	118	3,058	2.11	208	1,537	1.78	88
Sulphide Lower		332	8.12	87	8,022	6.27	1,617	3,132	5.57	560
Total Sulphide		**1,816**	**3.51**	**205**	**11,080**	**5.12**	**1,825**	**4,669**	**4.32**	**648**
Total Oxide		**270**	**1.47**	**13**	**1,870**	**1.32**	**80**	**404**	**1.27**	**16**
Total Oxide & Sulphide		**2,086**	**3.25**	**218**	**12,950**	**4.57**	**1,904**	**5,073**	**4.08**	**665**

Classification	Measured			Indicated			Inferred		
	Tonnes (kt)	Grade (g/t Au)	Insitu Gold (kOz)	Tonnes (kt)	Grade (g/t Au)	Insitu Gold (kOz)	Tonnes (kt)	Grade (g/t Au)	Insitu Gold (kOz)
Residues									
CIL	**571**	**7.83**	**144**	**0**	**0.00**	**0**	**0**	**0.00**	**0**
Total	**571**	**7.83**	**144**	**0**	**0.00**	**0**	**0**	**0.00**	**0**

Notes:

For the Mineral Resource estimate, the Qualified Person is Troy Fuller. The Mineral Resources are reported inclusive of Mineral Reserves.

Lower cut-off grades applied are 0.7 g/t Au for oxide, 1.0 g/t Au for near-surface sulphide (above 5050mRL) and 3.0g/t Au for underground sulphide mineralization (below 5050mRL).

CIL residues are stated as contained ounces - 25% recovery is expected. Recoveries are based on operating performances.

Mineral Resources are rounded to 1,000 tonnes, 0.01 g/t Au and 1,000 ounces. Minor discrepancies in summation may occur due to rounding.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The Mineral Resource estimate used a gold price of AUD$1500 per ounce.

The reported Mineral Resources are largely the result of work undertaken in December 2015 and reported by Newmarket Gold under Canadian reporting requirements in accordance with the NI43-101.

In all cases, the Qualified Person has reconciled the estimates to CIM standards as prescribed by NI 43-101.

The Authors are not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors that would materially affect the Mineral Resource estimate.

For further discussion on other issues that may impact upon the Mineral Resource estimates such as the reader is referred to Section 15, 19, 20 and 21 of this report.

The location and extents of the block models for each of these areas are displayed in Figure 14-1 below. Current underground mining activities are confined to the Central (Northern, Phoenix and Central Models) and Harrier (Harrier Model) Areas. Open pit mining activities were last undertaken in 2012 in the Robbin's Hill Area (Robbin's Hill Model). These areas will be described in detail below, as will descriptions of the Northern (Fosterville-Hunts Model) and the Southern (Southern Model) areas.

However, much of the data collection and systems developed for the Central and Harrier Areas were also applied to the Fosterville-Hunts Area, Southern Area and Robbin' s Hill Areas and so for brevity, the discussions on the latter three areas will highlight the geology of each area and differences from the Central/Harrier Area modeling.

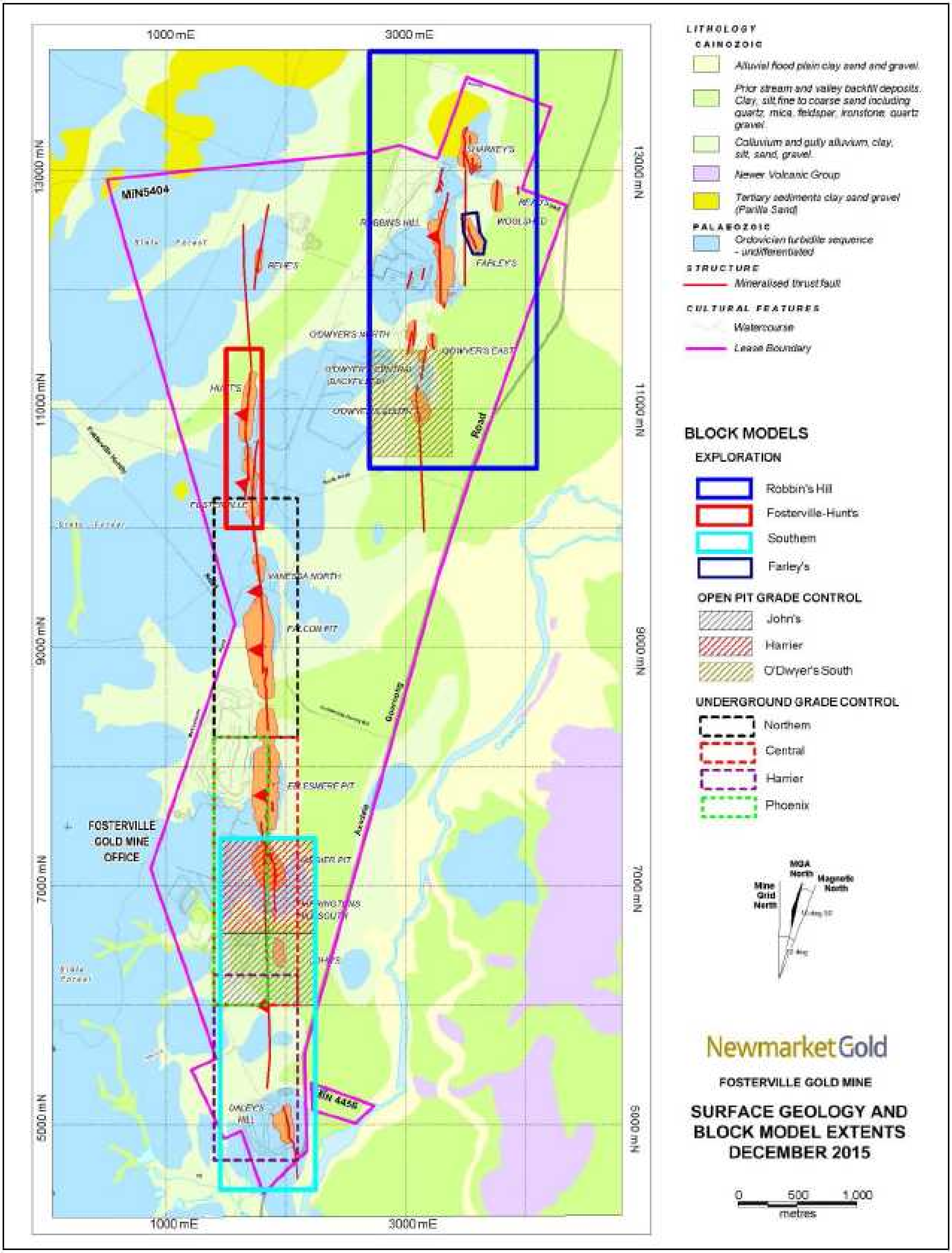

Figure 14-1 Plan showing mining lease and the area covered by each of the block models.

14.1 Central Area

14.1.1 Area Geology

The Central Area is defined as the zone between 6,050mN and 10,250mN (Mine Grid) encompassing both the Central, Phoenix and Northern underground models. The Northern Underground model extends from 8,250mN to 10,250mN with the Central and Phoenix models existing between 6050mN and 8250mN. The Phoenix and Central models are differentiated by a break at the 4600mRL where the Central Model encompasses domains from 4600mRL to surface and the Phoenix Model from 4600mRL to 4000mRL. Within the Central Area, there exist 8 current and 6 remnant mineralized zones:

Current	Remnant
Phoenix	Falcon
Lower Phoenix	Ellesmere
Lower Phoenix Footwall	Shamrock
Eagle	Robin
East Dippers	Griffon
Allwood	Vulture
Kestrel	
Splays	

Throughout 2015 the majority of drilling, mining, mapping, interpretation and subsequent mineral resource modelling were undertaken within the extents of the Lower Central Area, below the 5050mRL. The Mineral Resources in the Lower Central Area are detailed in Table 14-2.

Table 14-2 Central Area Lower Sulphide Mineral Resources (Inclusive of Mineral Reserves) below 5050mRL - Fosterville as at December 2015

Central Area Lower Sulphide Mineral Resources (Inclusive of Mineral Reserves) below 5050mRL - Fosterville as at December 2015									
Classification	Measured			Indicated			Inferred		
	Tonnes (kt)	Grade (g/t Au)	Insitu Gold (kOz)	Tonnes (kt)	Grade (g/t Au)	Insitu Gold (kOz)	Tonnes (kt)	Grade (g/t Au)	Insitu Gold (kOz)
Allwood	5	5.59	1	110	6.30	22	170	6.48	36
Eagle	23	16.76	12	178	10.97	63	43	27.21	37
East Dippers	1	6.85	0	544	9.79	166	27	16.12	14
Ellesmere	-	-	-	331	5.73	61	20	3.39	2
Harrier	-	-	-	48	3.96	6	25	3.62	3
Kestrel	6	6.69	1	960	4.70	145	175	5.13	29
Lower Phoenix	64	7.68	16	495	8.75	139	-	-	-
Lower Phoenix FW	37	10.38	12	278	8.16	73	34	4.89	5
Phoenix	151	7.58	37	627	6.54	132	59	4.89	9
Raven	-	-	-	119	8.12	31	-	-	-
Robin	-	-	-	68	8.39	18	-	-	-
Splays	-	-	-	912	5.74	169	298	3.98	38
Vulture	-	-	-	517	5.04	84	635	4.56	93
Stockpile*	27	4.65	4	-	-	-	-	-	-
Total Sulphide	**315**	**8.29**	**84**	**5,188**	**6.65**	**1,109**	**1,488**	**5.58**	**267**

Notes:

For the Mineral Resource estimate, the Qualified Person is Troy Fuller.

The Mineral Resources reported are inclusive of the Mineral Reserves for the same area.

A lower cut-off grade of 3.0 g/t Au is applied to Lower Sulphide Mineral Resources below 5050mRL.

Mineral Resources are rounded to 1,000 tonnes, 0.01 g/t Au and 1,000 ounces. Minor discrepancies in summation may occur due to rounding.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The Mineral Resource estimate used a gold price of AUD$1500 per ounce.

*Stockpile Inventory includes Lower Central Area Mineral Resources contained within the Run of Mine Stockpile and Coarse Ore Stockpile as at 31st December 2015.

14.1.2 Geological Models

In order to constrain the mineral resource models, a number of three-dimensional geological models were generated for each zone using MineSight software. The models produced were of three types:

- structural wireframe models
- mineralization wireframe models
- waste wireframe models

Structural models contain three-dimensional wireframe surfaces of major faults and minor structures as interpreted from surveyed data points obtained from open pit and underground mapping and diamond drill core logs. The mineralization model defines the interpreted gold-bearing mineralized envelopes and is constrained either by structural, lithological or grade boundaries. The waste model is defined by a 10-15 metre envelope surrounding the mineralization model.

Mineralization domain wireframes are constructed by the creation of solid polygons with sections that are clipped to the corresponding minimum drill spacing. This has resulted in interpretations completed on 6.25m sections in areas of open pit grade control drilling and on 25m in areas of underground grade control drilling and, 50m and 100m sections where there is only surface and underground exploration drilling.

Mineralization used within the domain boundary is selected based on a current cut off of 4 gram-metres (generally 2m at 2.0 g/t Au). Mineralization below the cut off may exist within the mineralized domain if there is contradictory data directly adjacent or if the intercept lays central to other peripheral economic intercepts on the same interpreted structure.

Data points that satisfy particular economic or geological criteria for inclusion are directly clipped into the domain solid so that the assay interval is either entirely within or entirely excluded from the interpreted mineralized envelope. Separate mineralization envelopes are created to distinguish between geologically or economically distinct zones such as high-grade/low-grade envelopes or changes in structural orientations.

Information derived from RC and diamond drill data (assays, structure, lithology, etc.) are used in the initial construction of the mineralized domains. Mineralized zones that become viable for mining are further constrained by the addition of geological mapping, surveyed structures, open pit blast hole samples, underground sludge hole and face samples (Figure 14-2).

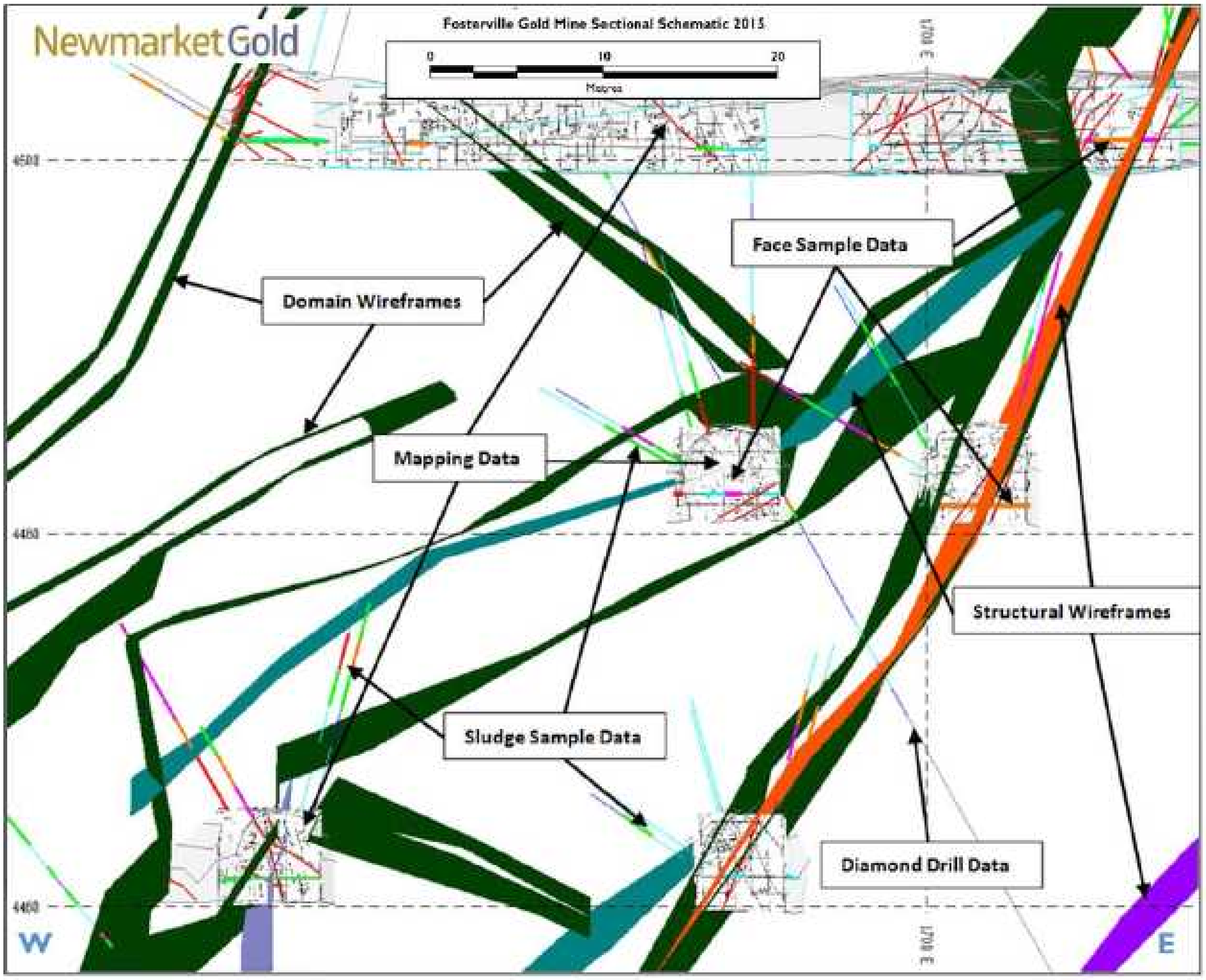

Figure 14-2 Section showing data for creating mineralization domain wireframes (underground).

14.1.3 Domains

Based on observed variations in geology, orientation, variography, geochemistry, statistics and spatial location within the Fosterville mine area, mineralization in the Central Area (Figure 14-2) has been divided into nineteen distinct domains, three outlier domains, one redundant and one common domain shared with the Harrier Area.

Table 14-3 Model domains, codes and assigned Mineralized Zones.

Domain Classification			
Model	**Domain Name**	**Domain Code**	**Mineralized Zone**
Central	**Fosterville HG**	1	Falcon, Vulture, Ellesmere
	Fosterville LG	2	Falcon, Vulture, Ellesmere
	Phoenix HG	3	Phoenix
	Phoenix LG	4	Phoenix
	Splay HG	5	Splays
	Splay LG	6	Splays
	Kite	7	Splays
	Kite LG (redundant)	*8*	*Splays*
	Raven (redundant)	*9*	*Raven*
	Vulture	10	Vulture
	Harrier OP	11	Harrier
	Phoenix Base	12	Phoenix
	East Dippers	18	East Dipper
Phoenix	**Audax**	1	Eagle
	Phoenix HG	3	Phoenix
	Splay HG	5	Splays
	Splay LG	6	Splays
	Allwood	8	Allwood
	Vertical	9	East Dipper
	Benu W1	10	Lower Phoenix Footwall
	Phoenix Base	15	Phoenix
	Benu	13	Lower Phoenix
	Benu FW	14	Lower Phoenix Footwall
	Kestrel	15	Kestrel
	Bedded East	16	East Dipper, Kestrel
	Shallow East Dippers	17	East Dipper
	East Dippers	18	East Dipper, Eagle
	Sphinx	19	Lower Phoenix Footwall
	Eagle	20	Eagle
Northern	**Fosterville HG**	1	Falcon, Vulture, Ellesmere
	Fosterville LG	2	Falcon, Vulture, Ellesmere
	Phoenix HG	3	Phoenix
	Splay LG	6	Splays
	Griffon	7	Splays

Domains are created due to the identification of a unique set of parameters that are coincident with economic mineralization traced through a number of drilled sections. Unique parameters may include the presence of a defining structure (Fosterville Fault, Phoenix Fault, Benu Fault etc.), consistent orientation along strike and dip, mineralization style (disseminated sulphide, massive stibnite or visible gold), spatial location or geological setting (hinge, oblique/oblique, parallel/parallel, parallel/oblique, oblique/parallel, etc.). Surrounding all the ore domains is a waste domain that was used to generate the waste gold grades in the immediate vicinity of the mineralization.

Broader zones of mineralization have been defined in the Central Area and each of these zones may consist of multiple domains (Table 14-3). Below are descriptions of the mineralized zones within the Central Area.

Phoenix

The Phoenix Mineralized Zone is situated within offset zones of Phoenix Syncline Hinge created by faulting within the Phoenix Shale package. Faulting that occurs at the top of ~8m moderately sericitised shale package is defined as the Phoenix Fault, with the Phoenix Base Fault occurring towards the base before transition into undifferentiated sandstones.

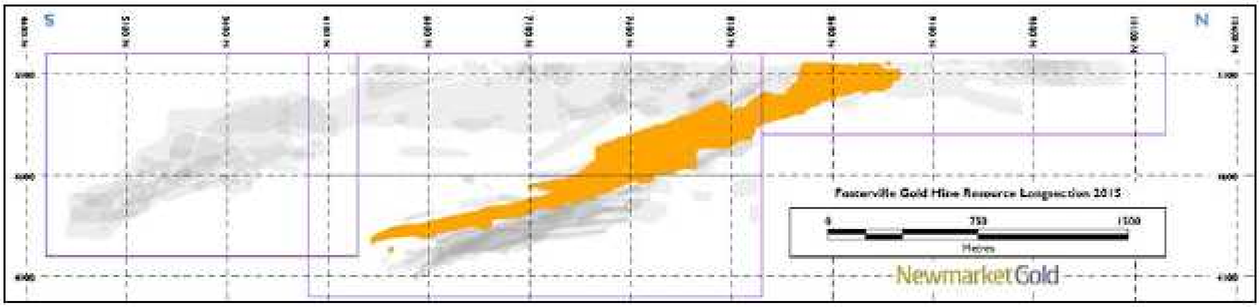

Figure 14-3 Longitudinal Projection of the Phoenix Mineralized Zone.

Movement and fluid generation for the Phoenix Fault appears to nucleate from the Fosterville Anticline as west dipping faulting branches through east dipping beds. This fault movement creates an offset of the syncline hinge resulting in wallrock brecciation and permeation of mineralized fluids into the surrounding country rocks. Brecciation and economic mineralization appear to cease as the system encounters the hangingwall offset of the syncline hinge sending the fluid into parallel bedding and limiting sulphide dissemination.

The mineralization in the Phoenix Domain plunges 15° to 20° to the south. Mineralization on the Phoenix Fault is consistent in width and geometry dipping 45° to 65° to the west with an internal high-grade shoot geometry that plunges roughly 70° to the south with a strike length of 30m to 40m and a width up to 20m.

The high-grade shoot geometry, believed to be related to subtle strike changes to the Phoenix Fault, appears also to be periodic in occurrence with a shoot occurring around every 200m between 7300mN and 8200mN. Syncline offset on the Phoenix Fault ceases around 7085.5mN with movement and mineralization transferring to Phoenix Base Fault from the 8212.5mN section becoming more evident from 7537.5mN (Figure 14-3).

Mineralization associated with the Phoenix Base and Phoenix Footwall Faults occurs south of 7337.5mN and remains open down plunge. The Phoenix Base area differs slightly to the Phoenix as fluid flow and fault movement appear to be related to compressive compensation of the Phoenix Syncline Hinge along the Kestrel Shale package. Current faulting mechanisms suggest that as the Phoenix Syncline Hinge is squeezed by east-west regional compression, a pervasive low angle structure (~35^{o}) links from the eastern limb of the Kestrel Shale package across to the Phoenix Base laminated quartz vein with ~30m of movement at its maximum. Sulphide mineralization appears to be sourced from migration up the Phoenix Syncline Hinge; however, the lack of antimony within the zone could point to down plunge controls that have not yet been identified with diamond drilling.

Lower Phoenix

The Lower Phoenix Mineralized Zone encompasses mineralization that is directly related to the west dipping faulting associated with the Benu sedimentary strata package below 4500mRL. Source mineralization is interpreted to migrate up the system from deep intersections with other mineralized structures including potentially hinges and other proximal oblique structures. Fluids utilize fault and fracture pathways to migrate up plunge and dip towards the Fosterville Anticline before linking across to a zone of distributed faults, which eventually re-forms up dip into the Phoenix.

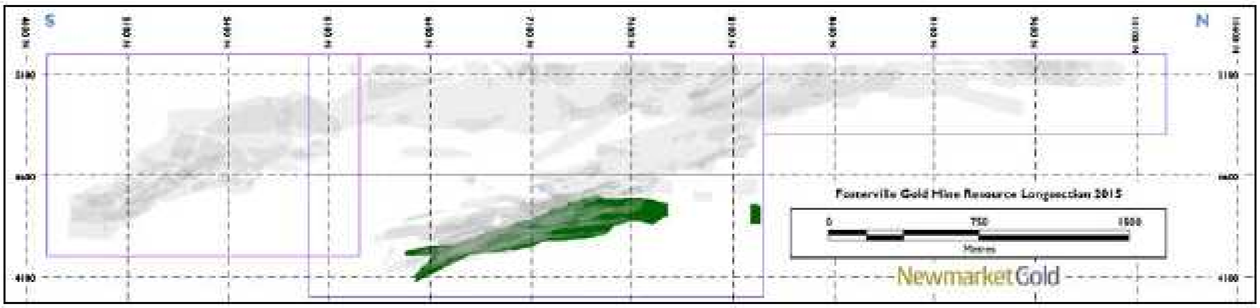

Figure 14-4 Longitudinal Projection of the Lower Phoenix Mineralized Zone.

The Lower Phoenix is defined by west dipping faulting on the Benu Shale sequence where anticline thrust displacement of ~80m occurs. Components of mineralization can also be traced up dip into east dipping stratigraphy and down dip into parallel bedded zones giving a maximum dip extent of 190m. The system currently remains open to the north and south so maximum plunge extent has not yet been characterized.

The system orientation is predominately controlled by west dipping bedding orientation giving the zone a similar structural orientation to that of both Phoenix and Falcon zones with a strike of ~355°, a general plunge of ~20°S and a dip of 55°W in parallel/oblique settings, but shallowing to 35°W dip in oblique/oblique settings.

To the south of the Lower Phoenix, mineralization is strongly influenced by the intersection with the Eagle system where faulting appears to cross cut west dipping bedding strata providing an environment where parallel / parallel economic mineralization occurs to the north and up plunge of this intersection.

Extension drilling programs are planned to test up and down plunge components of the ore zone, which presently remain unconstrained by drill data.

Lower Phoenix Footwall

The Lower Phoenix Footwall Mineralized Zone encompasses mineralization that is associated with west dipping structures footwall to the Lower Phoenix system below 4500mRL. Mineralization is interpreted to utilize similar networks to those utilized by the Lower Phoenix system. Clearly discernable sedimentary horizons such as the Pelican Shale package are evolving as the next step in fluid migration pathways however non bedding conformable systems are evident that require further research.

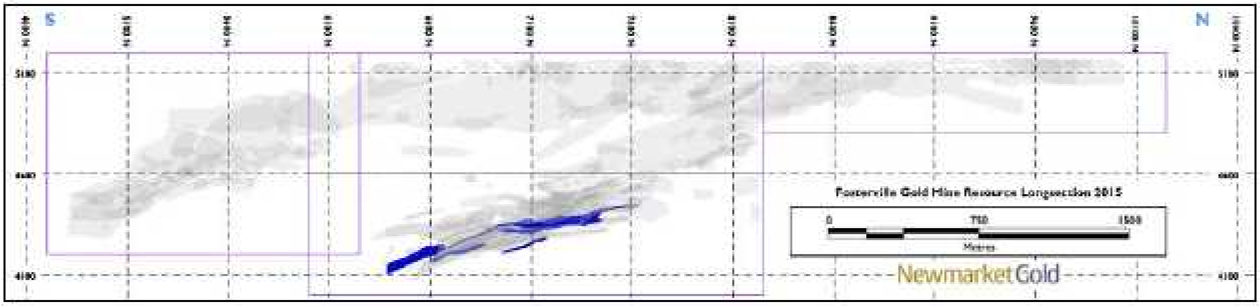

Figure 14-5 Longitudinal Projection of the Lower Phoenix Mineralized Zone.

Mineralization domained within the Lower Phoenix FW is interpreted to be due to low angled structures that have minimal offset but have been hydraulically fractured by gold bearing fluids. The hydraulically fractured zones can create large quartz carbonate veins that can be several meters wide in true thickness. The presence of a number of laminations within the quartz veins indicates a series of events with differing geochemistry including later stage quartz-stibnite mineralization and visible gold (Figure 14-5).

The vein systems are interpreted to migrate across east dipping stratigraphy appearing to terminate on prominent stratigraphic units such as the Kestrel East, Pegasus East and Allwood East LQ veins. The termination is due to mineralizing fluids moving out of an oblique/ oblique setting as the structure cuts across beds into a parallel/parallel setting as fluids escape into the east dipping bedding parallel laminations.

East Dippers

The East Dippers system has developed at depth as the Fosterville Anticline has diverged away from the Phoenix Syncline system creating new networks for fluids to migrate up the Fosterville system. Systems utilize similar mechanics to that established within the west dipping fault network where rheological contrasts between bedding units (primarily slip associated with graphitic laminated quartz veins around carbonaceous shales) provide an accommodation zone for stress and mineralization. (Figure 14-6).

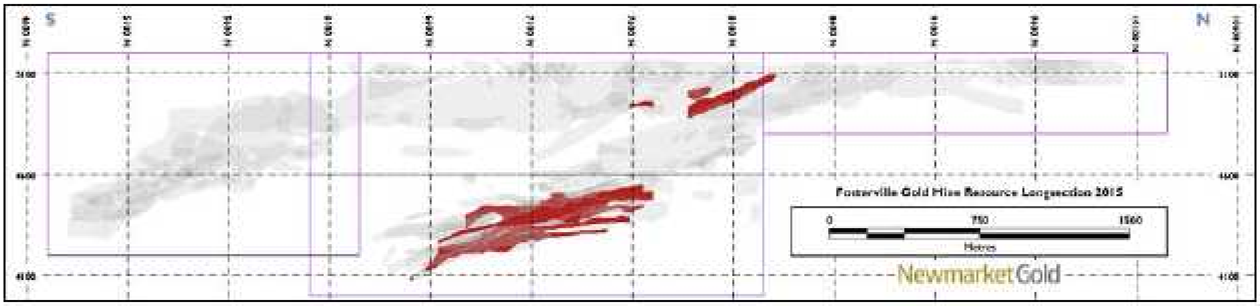

Figure 14-6 Longitudinal Projection of the East Dippers Mineralized Zone.

The difference between the west and east dipping packages are in the way that sedimentary packages accommodate forces acting on the zone. The east dipping zones accommodate stress by attenuation where the more ductile shale package deforms plastically whereas the more sand rich units show shows of brittle deformation in the form of ladder veins. These veins sets that radiate out from the shale boundaries perpendicular to the bedding orientation provide a mechanism for sulphides to leach into the host rocks.

The environments where East Dipper system occur have shale packages that correlate to a west dipping counterpart such as Kestrel, Allwood, Benu and Pegasus Zones. The east dipping fault naming convention utilizes the identified shale characteristics matched to the west dipping counterpart and given the E suffix to denote the east dipping status of the structure.

Eagle

The Eagle system occurs below 4400mRL where forces look to accommodate strain between the Fosterville Anticline and Phoenix Syncline via east dipping structures that are discordant to bedding. Although similar to the East Dippers System in spatial locality, Eagle differs as east dipping faults link from one east dipping shale package to another where the bedding angle is high (>70^{o}). This movement from across bedding creates a fault angle oblique to bedding that allows for mineralization to permeate into the host rocks (Figure 14-7).

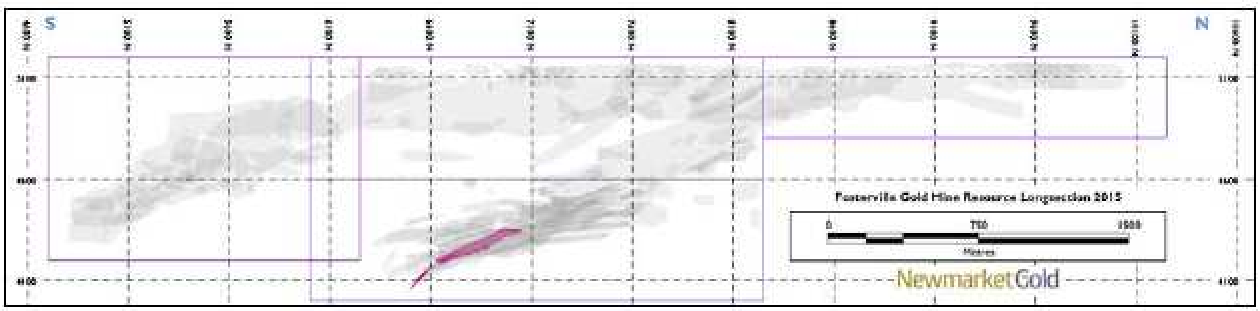

Figure 14-7 Longitudinal Projection of the Eagle Mineralized Zone.

Movement on the system via direct underground measurement and sedimentary horizon displacement appears to have a sinistral strike slip orientation. Predominant slip orientations on west dipping structures indicate a steep dip slip movement with a plunge to the south. How the sinistral movement fits into the overall Fosterville system architecture is under evaluation.

Figure 14-8 Wall mapping on the P4310 level convergence of the D20 Eagle and D14 Benu FW

Mineralization gold grades on the Eagle system increase up-dip where east dipping faulting is proximal to the Fosterville Anticline and west dipping faulting. The convergence of east and west dipping structures in proximity to the Fosterville Anticline appears to provide a barrier for fluid migration resulting in flow textures of quartz and stibnite (Figure 14-8). Isolated areas of visible gold can be seen within the zone as fine specks that form in alignment with stylolitic fractures that can extend for up to ~10cm. Typically the arsenopyrite / pyrite mineralization within the zone is weaker with grades in the 1-2 g/t Au range with sulphide dissemination localized around the zone.

Moving down-dip away from the hinge, the quartz stibnite vein pinches out with disseminated arsenopyrite and pyrite increasing in intensity and grade. Dissemination is still localized to the main Eagle Fault (with 1-2m of the structure), however, interaction with bedded faults creates zones where fracture interplay between the two systems increases the fluid flux and therefore increases the economic width of the zone.

Down-plunge and dip continuation of the Eagle system is currently being evaluated, however, intercepts that show potential extensions to the system, have already been intersected.

Splays

Throughout the Central Area, there are a number of significant mineralized structures and settings that fail to have size, confidence or spatial continuity to develop into extensive mineralized zones These systems are captured within the Splays HG and Splays LG domain and present either proximal mining opportunity or future potential growth prospects. Most systems within the Splay domains are defined by shallow west dipping faulting (~30-40°), of anastomosing nature and highly variable grade distribution (Figure 14-9).

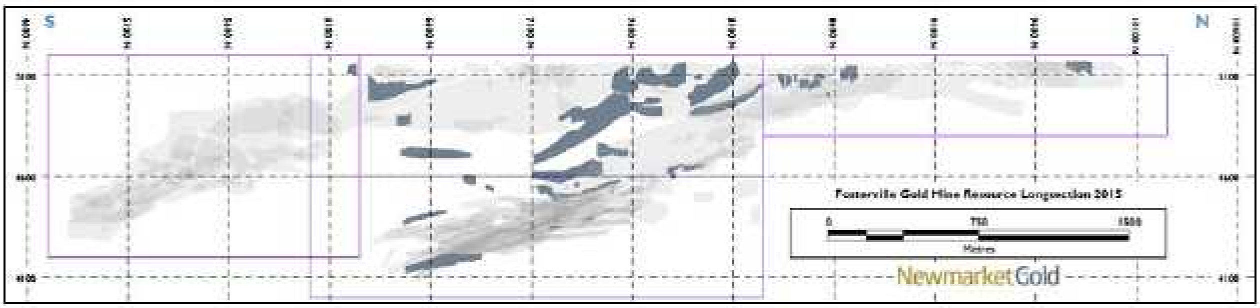

Figure 14-9 Longitudinal Projection of the Splays Mineralized Zone.

Splay faults are interpreted to be short lived structures that split as the structure moves from a Lower Phoenix Zone setting across to the Phoenix Zone setting. Larger splay faults are prevalent between the anticline offset of the Fosterville Fault, where the large thrust movement has nucleated a number of smaller structures. The largest of the splays in the zone is the Kite Fault. The Kite area of mineralization is interpreted to be due to an oblique/oblique setting created between bedding relationships with the Kite Fault. The Kite Fault is an example of a mid-splay system that nucleates from the Fosterville Fault linking across to the Phoenix Footwall Syncline setting. The main system extends from 7662.5mN through to 7062.5mN with an overall dip of ~30° to the west and a plunge of 25° to the south.

Allwood Domain

The Allwood area is interpreted to be created by 30m of fault movement along the Allwood Shale package that offsets an anticline creating a parallel/oblique setting for mineralization. The system is analogous to the Phoenix Lower Zone setting and extends 562m to the south from the 7675mN section. Orientation of the Allwood Zone is similar to other geometries constrained by a west dipping hangingwall (Fosterville HG, Fosterville LG, and Benu) with a 65° dip to the west and a 10° plunge to the south. Assays, composites and blocks occurring within the Allwood are coded into the Allwood (DOMAIN=8) domain (Figure 14-10).

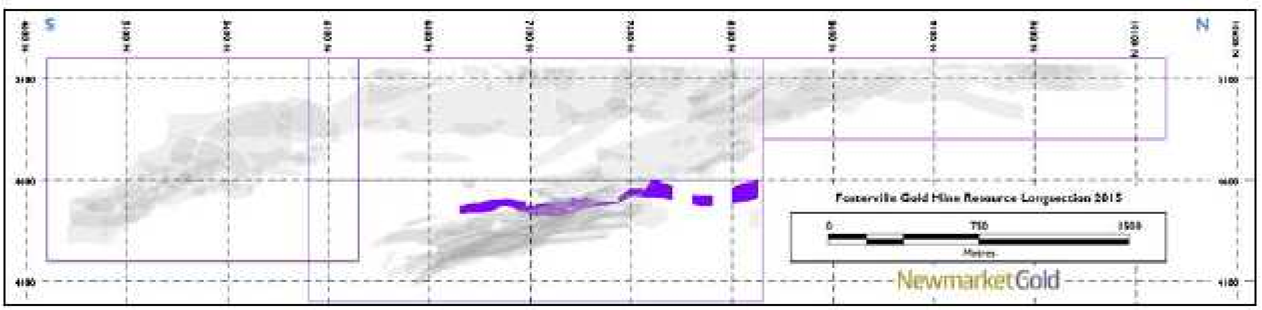

Figure 14-10 Longitudinal Projection of the Allwood Mineralized Zone.

Kestrel

During 2015, further targeting of the Kestrel System was undertaken to establish the pervasiveness and continuity of high grade mineralized intercepts. Results consistently showed a zone of broad (+6m) heterogeneous low to moderate (1 – 4 g/t) sulphide mineralization with interspersed high grade assays. The geology department investigated the system to see if there was an element of continuity to the high grade intersections but could not establish a confident set of definable characteristics to correlate high-grade gold along strike and down dip (Figure 14-11).

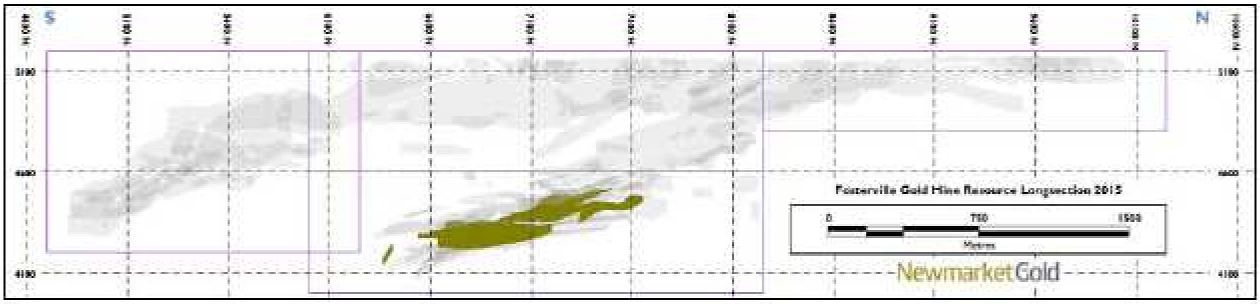

Figure 14-11 Longitudinal Projection of the Kestrel Mineralized Zone.

The work conducted established that there are some common elements with mineralization distribution through the zone. The intercepts typically were broad zones of mineralization centralized around the Phoenix Syncline Hinge. Fluid pathways appeared to utilize weaknesses in cleavage and flexural slip planes between contrasting beds. Vein intensity and direction appears to be correlatable to rock type as sand rich packages exhibit brittle radial fracture sets more aligned with hinge compressive stresses rather than brittle fault offset. Grade variability appears to be related to application of the varying conditions of geochemistry, rock permeability, rheology and fluid flux through the zone and therefor difficult to align into discreet domains.

Falcon

The Falcon Mineralized Zone is situated on the Fosterville Fault where it displaces the Fosterville Anticline along a distinguishable black shale horizon. The thrust movement on the fault creates an offset of ~500m with several splay faults that nucleate from the main Fosterville Thrust. These splays cross east dipping bedding creating smaller orebodies such as the Ellesmere and Vulture Mineralized Zones. (Figure 14-12).

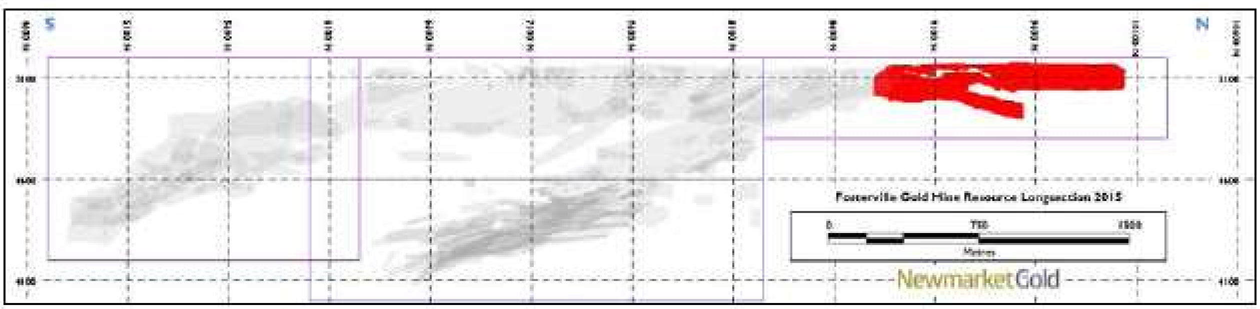

Figure 14-12 Longitudinal Projection of the Falcon Mineralized Zone.

The Falcon Mineralized Zone consists of Fosterville HG (DOMAIN=1) and Fosterville LG (DOMAIN=2) domains. Fosterville HG is reasoned to be a population of discernibly higher grade assays that exist due to a shoot geometry that is geologically controlled within the larger Fosterville LG domain. A plunge reversal occurs between 8800mN and 8900mN and all of the mineralization between 8900mN and 11000mN plunges gently to the north. The vast majority of the mineralization in the Falcon domain occurs on the Fosterville Fault and dips about 70° to the west. Most of this domain is relatively shallow (less than 150m below surface) and has been drilled by either RC drilling grade control drilling on 6.25m spaced sections or by RC and diamond exploration drilling on 20m spaced sections.

Ellesmere

The Ellesmere Mineralized is characterized by Fosterville Zone mineralization and resides primarily between the Fosterville HG and Fosterville LG domains south of the culmination. Overall the plunge of the mineralization within the Ellesmere Orebody appears to be 20° to 40° to the south with internal narrow (~20m) high-grade shoots plunging 70°S and occurring at roughly 100m intervals. The high-grade shoots are believed to be the results of smaller footwall splay fault interaction with the Fosterville Fault. Mining of the Ellesmere Orebody was completed in 2010 (Figure 14-13).

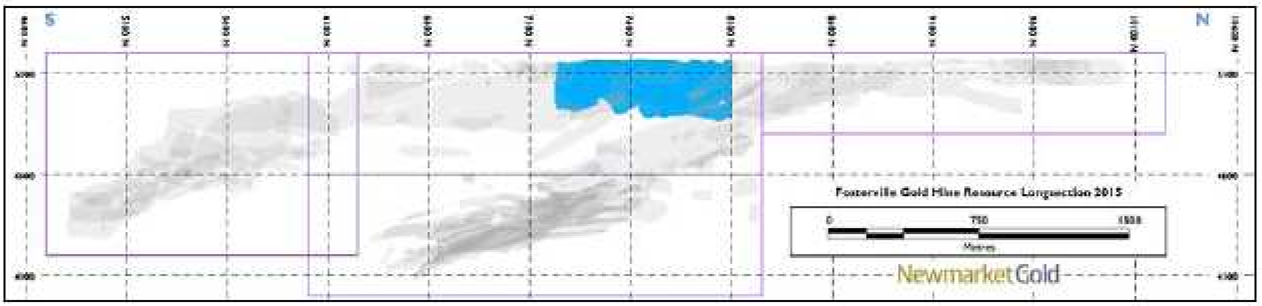

Figure 14-13 Longitudinal Projection of the Ellesmere Mineralized Zone.

Raven

The Raven Mineralized Zone exists as a zone of high-grade splay mineralization north of the Phoenix Mineralized Zone analogous with Phoenix Zone mineralization. The orebody is situated where fault movement associated with the Phoenix Fault links across to the Phoenix Base Footwall Syncline Hinge moving into an oblique/oblique setting. Mineralization forms on a number of splay structures that typically have a shallower dip (~40°) and strikes more NNW than the typical N-S bearing of the Phoenix Zone (Figure 14-14).

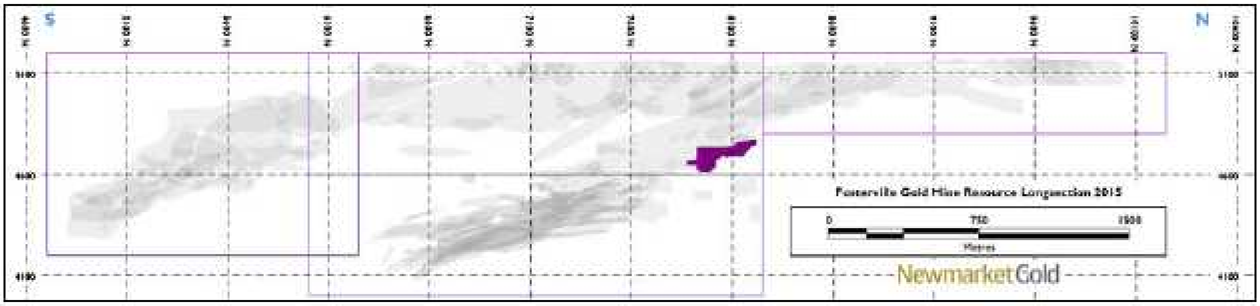

Figure 14-14 Longitudinal Projection of the Raven Mineralized Zone.

Vulture

The Vulture Mineralized Zone occurs between 6262.5mN and 7337.5mN in a zone characterized primarily by Harrier faulting where economic mineralization occurs proximal to the intersection between the interpreted Harrier Base Fault and the Fosterville Fault. The main Vulture Mineralized Zone on the Harrier Base Fault dips ~45°W, steepening as the fault diverges from the Fosterville Fault. Mining of parts the Vulture Zone was completed in early 2012. However, subsequent knowledge gained from mining the Harrier Zone is being applied to the remainder of the Vulture Zone to optimize further extraction potential (Figure 14-15).

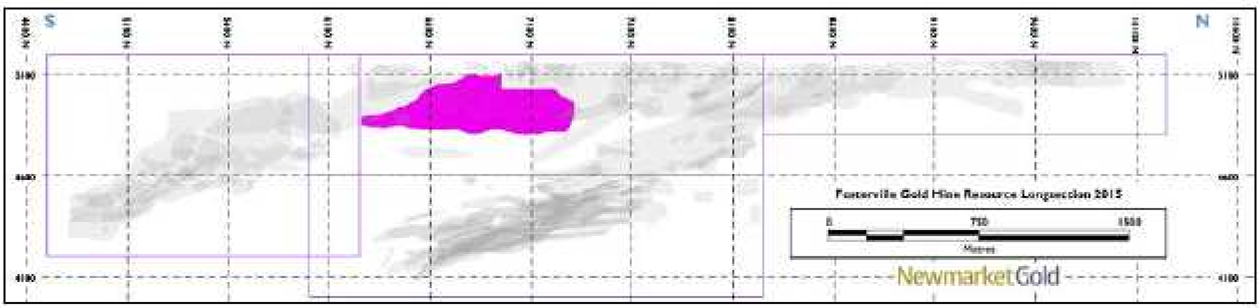

Figure 14-15 Longitudinal Projection of the Vulture Mineralized Zone.

Robin, Griffon & Shamrock

The Robin Mineralized Zone, shown in pink in Figure 14-16 is interpreted to be a zone where mineralization switches back from the Phoenix Fault across to the Fosterville Fault around the hangingwall section of the Phoenix Syncline Hinge. The fault network is a combination of east and west dipping structure that has a zonation plunge on the intersection with the Fosterville Fault of ~30^{o}. The interaction Zone occurs from 8600mN to 8100mN where separation distance between the Fosterville and Phoenix Faults widens to the south reducing the intensity of faulting reducing mineralization intensity.

The Griffon Mineralized Zone, shown in green (Figure 14-16), is a zone of mineralization on the Phoenix Keel Zone where faulting from the Fosterville Fault directly links across to the footwall section of the Phoenix Syncline Hinge. The zone exists between 8800mN and 8600mN with mining completed in 2009.

The Shamrock Mineralized Zone, shown in teal (Figure 14-16), is a Zone of mineralization footwall to the Fosterville Fault where the Phoenix Fault is directly adjacent to the system. The zone existed between 8600mN and 8350mN with mining completed in 2009.

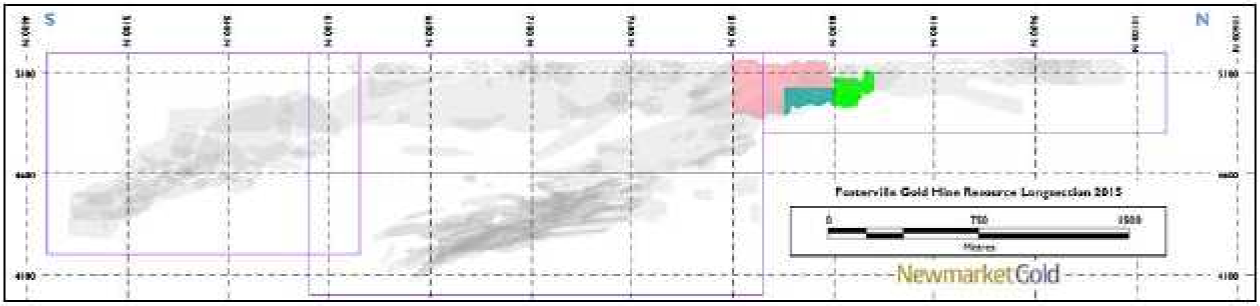

Figure 14-16 Longitudinal Projection of the Robin (pink), Griffon (green), and Shamrock (teal) Mineralized Zones.

14.1.4 Drilling Data

After the geological models and subsequent mineralization domains have been defined, the drill hole assay data used to produce the model was subjected to a number of data preparation processes:

1. Files containing all drillhole logging and assay data were imported from the acQuire production and exploration database into MineSight using an automated script.

2. A MineSight procedure coded the drill holes with the appropriate properties from the geological models and a drill hole composite file was constructed for values inside the mineralization wireframes.

3. The files were viewed in MineSight in order to identify holes that contained obvious erroneous data missed during the validation process. Data that was considered erroneous was either corrected or deleted from the data set. Note: step 1 and 2 were also done prior to the geological models being finalized to ensure the interpretations were finalized on a validated drill hole file.

In combination, the Dec 2015 drill hole files used for the Central Area Models (1512_PRM, 1506_CRM and 1201_NRM) contained a total of 5,700 drill holes between them to estimate mineralization, of which 3195 (56.1%) are RC and 2505 (43.9%) diamond core. There are a number of holes that appear both in the Central Model and the Northern Model. This is to allow for smoothness of interpolation from one model boundary to another (Table 14-4).

Table 14-4 Central Area Resource Model Drilling Data Extents

Central Area Resource Models Drilling Data Extents									
Model	North Min (m)	North Max (m)	RL Min (m)	RL Max (m)	Total Holes	Diamond Holes	RC/AC Holes	% Diamond Holes	(% RC/AC) Holes
1506_CRM	6000	8250	4600	5200	2914	1070	1844	36.7	63.3
1512_PRM	5800	8450	3950	4600	1041	1041	0	100.0	0.0
1201_NRM	8250	10250			1745	394	1351	22.6	77.4
					5700	2505	3195	43.9	56.1

Compositing

The raw sample results were composited to 2m intervals in the 1512_PRM, the 1506_CRM models and the 1201_NRM (Northern) Model using the MineSight compositing procedure. A 2m composite length was selected as it encompasses a high proportion of RC drill assay data, of which a high proportion is of 2m length or greater. Further work is required to ascertain the increasing proportion of recent ~1m underground diamond assay data collated as the mine reaches depths unobtainable to surface RC drilling. In 2015 the Central area model was divided into two between 5800mN and 8250mN on the 4600mRL. This has achieved a faster running time for the model updates required in the Lower Phoenix Zone below the 4600mRL.

The compositing process creates 2 or 1m composites of the primary assay intervals in a downhole direction honoring the coded geological domains. The MineSight software downhole compositing routine provides an option to accumulate short intervals (up to 50% of the composite length) into the preceding interval. Assay intervals above the minimum 50% primary sample length are treated as a unique composite interval. For example, an assay interval over 1.0m in length is left in the composite file as is, and an assay interval less than 1.0m is added into the preceding composite interval (Figure 14-17). This option has been used to prevent a number of smaller intervals from forming on the downhole margins of estimation domains, and as such all intervals can be used in the estimation process.

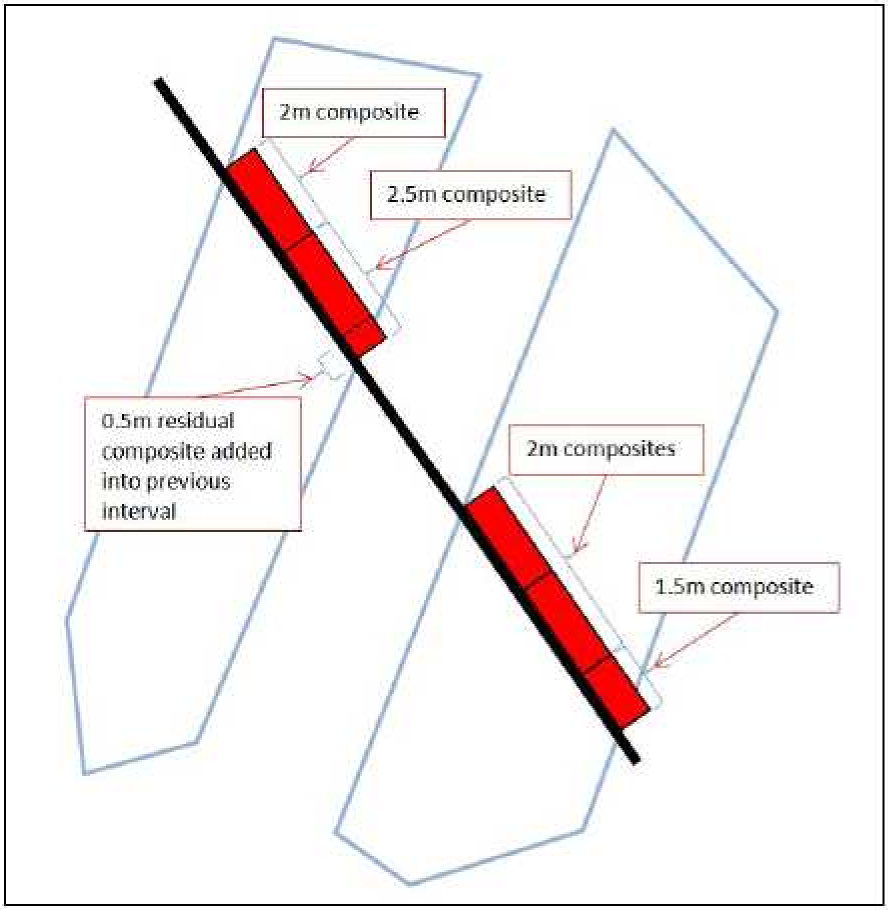

Figure 14-17 Downhole compositing where domain boundaries are honored in the composite file.

A listing of descriptive statistics for the estimated domains is provided for the Northern Model (1201_NRM) in Table 14-5.

Table 14-5 Descriptive statistics for the Northern Model.

Model	**1201_NRM**		**Descriptive Statistics**					
Date:	**Dec-2011**							
Variable	Data Type(s)	Number of Samples	Minimum	Maximum	Mean	Std Dev g/t	Variance g/t2	Coeff of Var
Code 1 Fosterville HG								
Au 2.0m Composites TC 40	DD	1,701	0.01	49.60	4.61	5.45	29.70	1.18
Code 2 Fosterville LG								
Au 2.0m Composites	DD	9,949	0.00	104.60	5.66	6.63	43.96	1.17
Code 3 Phoenix HG								
Au 2.0m Composites	DD	4,021	0.00	60.44	5.50	6.95	48.30	1.26
Code 6 Splay LG								
Au 2.0m Composites	DD	740	0.00	36.89	2.25	3.41	11.63	1.52
Code 7 Griffon								
Au 2.0m Composites	DD	101	0.20	57.21	9.74	10.62	112.78	1.09

A listing of composite statistics is provided in Table 14-6 for the Northern Model (1201_NRM).

Table 14-6 Composite statistics by composite length in the Northern Model.

Composite Length	Number	% of Composites	Mean Length (m)	Mean Grade (g/t Au)
< 1.0m	110	1%	0.24	3.37
≥ 1.0 and < 2.0	459	5%	1.44	5.49
≥ 2.0m	7,879	93%	2.01	5.39
Total	8,448	100%	1.96	5.37

A listing of descriptive statistics for the estimated domains is provided in Table 14-7 and Table 14-8 for the (1512_PRM) Phoenix Model and the 1506_CRM Central Models respectively. These statistics are provided as a context for the size and the average grade in each of the domains. The 1201_NRM model name encompasses the build date of the model and infers that the model includes the latest drilling and interpolation data in that respective area. Therefore the Northern area has not had interpretational and/or drilling additions since January 2012. Similarly the Central model has not had any changes since June 2015. However, the Phoenix model includes areas within the active mining and drilling zones and was completed in December 2015.

Table 14-7 Descriptive Statistics for the Phoenix Model.

Model:	**1512_PRM**		**Descriptive Statistics**					
Date:	**Dec-2015**							
Variable	Data Type(s)	Number of Samples	Minimum	Maximum	Mean	Std Dev (g/t Au)	Variance (g/t^2)	Coeff of Variation
Code 1 Audax								
Au Raw	DD	85	0.24	955.00	85.40	183.60	33,708.96	2.15
Au 2.0m Composites	DD	26	0.45	391.76	72.80	101.00	10,201.00	1.39
Code 3 Phoenix HG								
Au Raw	DD	526	0.02	75.00	7.99	7.79	60.68	0.97
Au 2.0m Composites	DD	236	0.53	47.98	7.71	5.68	32.26	0.74
Code 5 Splay HG								
Au Raw	DD	287	0.04	30.00	6.04	5.19	26.94	0.86
Au 2.0m Composites	DD	127	0.50	18.25	5.78	3.35	11.22	0.58
Code 6 Splay LG								
Au Raw	DD	904	0.01	514.80	6.76	21.19	448.97	3.13
Au 2.0m Composites	DD	437	0.01	514.80	6.40	19.41	376.75	3.03
Code 8 Allwood								
Au Raw	DD	219	0.06	33.30	6.01	5.28	27.88	0.88
Au 2.0m Composites	DD	97	0.73	26.39	6.00	3.98	15.84	0.66
Code 9 Vertical								
Au Raw	DD	840	0.01	7,368.00	35.78	281.15	79,045.32	7.86
Au 2.0m Composites	DD	283	0.01	1,115.37	27.31	92.33	8,524.83	3.38
Code 10 Benu W1								
Au Raw	DD	336	0.01	56.50	26.16	1.87	3.50	0.07
Au 2.0m Composites	DD	104	0.04	67.62	22.73	8.95	80.10	0.39
Code 12 Phoenix Base								
Au Raw	DD	802	0.01	1,694.70	11.37	61.82	3,821.71	5.44
Au 2.0m Composites	DD	324	0.01	806.87	11.21	47.41	2,247.71	4.23
Model:	**1512_PRM**		**Descriptive Statistics**					
Date:	**Dec-2015**							

Variable	Data Type(s)	Number of Samples	Minimum	Maximum	Mean	Std Dev g/t	Variance g/t2	Coeff of Var
Code 13 Benu								
Au Raw	DD	1884	0.01	7,707.00	17.53	6.17	38.07	0.35
Au 2.0m Composites	DD	735	0.04	71.63	13.62	3.76	14.14	0.28
Code 14 Benu FW								
Au Raw	DD	573	0.03	86.80	9.18	1.22	1.49	0.13
Au 2.0m Composites	DD	213	0.35	55.20	8.87	6.63	43.96	0.75
Code 15 Kestrel								
Au Raw	DD	768	0.01	25.20	3.92	3.04	9.24	0.78
Au 2.0m Composites	DD	318	0.01	19.21	3.85	2.21	4.88	0.57
Code 16 Bedding East								
Au Raw	DD	996	0.01	17.40	6.41	7.48	55.95	1.17
Au 2.0m Composites	DD	430	0.22	17.54	6.06	7.86	61.78	1.30
Code 17 Shallow East Dippers								
Au Raw	DD	63	0.27	22.80	5.58	4.60	21.16	0.82
Au 2.0m Composites	DD	29	1.45	11.79	5.17	2.73	7.45	0.53
Code 18 East Dipper								
Au Raw	DD	714	0.01	65.30	6.57	6.06	36.70	0.92
Au 2.0m Composites	DD	329	0.01	24.50	6.25	4.05	16.36	0.65
Code 19 Sphinx								
Au Raw	DD	72	0.03	90.70	8.52	12.32	151.73	1.45
Au 2.0m Composites	DD	28	1.69	44.78	8.53	8.62	74.30	1.01
Code 20 Eagle								
Au Raw	DD	778	0.02	17,050.00	48.74	643.81	414,491.32	13.21
Au 2.0m Composites	DD	276	0.20	1,667.77	27.56	123.73	15,309.06	4.80

Table 14-8 Descriptive Statistics for the Central Model

Model:	**1506_CRM**							
Date:	**Jun-2015**		**Descriptive Statistics**					
Variable	Data Type(s)	Number of Samples	Minimum	Maximum	Mean	Std Dev g/t	Variance g/t2	Coeff of Var
Code 1 Fosterville HG								
Au Raw	DD	571	0.02	72.00	7.93	6.45	41.60	0.81
Au 2.0m Composites	DD	287	0.02	28.98	7.49	4.68	21.90	0.62
Code 2 Fosterville LG								
Au Raw	DD	6,993	0.00	41.00	2.81	3.71	13.73	1.32
Au 2.0m Composites	DD	6,556	0.00	41.00	2.82	3.67	13.45	1.30
Code 3 Phoenix HG								
Au Raw	DD	2,694	0.01	104.80	8.37	8.67	75.20	1.04
Au 2.0m Composites	DD	1,175	0.01	49.54	7.96	6.39	40.81	0.80
Code 4 Phoenix LG								
Au Raw	DD	124	0.01	27.3	4.08	4.79	22.94	1.17
Au 2.0m Composites	DD	75	0.01	17.5	4.33	4.23	17.89	0.98
Code 5 Splay HG								
Au Raw	DD	873	0.01	57.60	6.41	7.01	49.10	1.09
Au 2.0m Composites	DD	394	0.01	38.18	6.11	5.55	30.80	0.91
Code 6 Splay LG								
Au Raw	DD	2,291	0.00	28.80	2.24	2.85	8.14	1.27
Au 2.0m Composites	DD	1,875	0.00	24.60	2.04	2.53	6.40	1.24
Code 7 Kite								
Au Raw	DD	298	0.42	28.60	8.02	5.96	35.52	0.74
Au 2.0m Composites	DD	145	1.21	23.85	7.73	4.39	19.27	0.57
Code 10 Vulture								
Au Raw	DD	595	0.14	24.20	5.03	2.86	7.84	0.56
Au 2.0m Composites	DD	313	0.45	19.9	4.97	2.35	5.52	0.47

Model:	**1506_CRM**							
Date:	**Jun-2015**		**Descriptive Statistics**					
Variable	Data Type(s)	Number of Samples	Minimum	Maximum	Mean	Std Dev g/t	Variance g/t2	Coeff of Var
Code 11 Harrier OP								
Au Raw	DD	1,635	0.00	15.33	2.44	2.69	7.24	1.10
Au 2.0m Composites	DD	1,574	0.00	15.33	2.41	2.66	7.08	1.10
Code 12 Phoenix Base								
Au Raw	DD	184	0.01	52.40	10.51	8.80	77.44	0.84
Au 2.0m Composites	DD	84	0.01	32.40	10.08	7.08	50.13	0.70
Code 18 East Dipper								
Au Raw	DD	245	0.06	59.40	7.52	6.50	42.25	0.86
Au 2.0m Composites	DD	114	0.32	24.61	7.25	4.06	16.48	0.56

A listing of composite statistics is provided in Table 14-9 for the (1512_PRM and 1506_CRM) Phoenix and Central Models.

Table 14-9 Composite statistics by composite length for the Central Model (1506_CRM) and Phoenix Model (1512_PRM).

Model	Composite Length	Number	% of Comps	Mean length (m)	Mean Grade (g/t Au)
1512_PRM	< 1.0m	461	11.8	0.66	8.09
	≥ 1.0 and <2.0m	1,082	27.6	1.38	9.19
	≥ 2.0m	2,372	60.6	2.08	12.75
	Total	**3,915**	**100.0**	**1.72**	**11.22**
1506_CRM	< 1.0m	86	2.1	0.68	4.92
	≥ 1.0 and <2.0m	572	14.0	1.34	7.14
	≥ 2.0m	3,431	83.9	2.03	0.56
	Total	**4,089**	**100.0**	**1.91**	**1.57**

Variography

Modeling of spatial continuity (variography) was carried out using MineSight software. Experimental variograms were generated from domain composites including new black shale domains. Sulphur is estimated in each domain as a variable using the domain geology shape with a general Sulphur variogram. Non-Carbonate Carbon (NCC) is estimated using two broad domain shapes, encompassing east and west geometries. Both geometries use a general variogram structure. Gold grade continuity is the highest along structures contained within parallel/oblique sedimentary host rock bedding contrasts. Within the parallel/oblique bedding zones it is common to see variogram structure ranges of up to 80m. In oblique/oblique host sedimentary settings the spatial grade continuity is less consistent, giving rise to variogram structures with ranges of less than 40m. Therefore high level mining decisions (reserve block and capital development) are made where drill spacing is at least 50m x 50m and a decision to mine a given level is only made on an indicated resource with a drill spacing of 25m x 25m (sulphide hosted gold resources only). A similar rationale currently exists for confidence around the development and extraction of the visible gold quartz hosted style mineralization.

Variogram parameters used for gold in the Northern Block Model (1201_NRM) estimation are listed inTable 14-10. Variogram parameters used for Sulphur and NCC in the block model estimation in the 1201_NRM are listed inTable 14-11.

Table 14-10 Variogram parameters used for Northern Model gold estimation.

GOLD VARIOGRAM PARAMETER TABLE																	
1201_NRM																	
AREA	Z Rotation	X Rotation	Y Rotation	Nugget	1st Rotation Spherical Sill	Range (y)	Range (x)	Range (z)	2nd Rotation Spherical Sill	Range (y)	Range (x)	Range (z)	3rd Rotation Spherical Sill	Range (y)	Range (x)	Range (z)	Total Varianc
	meds rotation																
D01 Fosterville LG	0	20	70	3.7	5.7	7	5	5	1.6	20	20	10	3.2	48	55	15	14.20
D02 Fosterville HG	0	20	70	3.7	5.7	7	5	5	1.6	20	20	10	3.2	48	55	15	14.20
D03 Phoenix HG	355	20	50	20.0	10.0	10	15	5	21	45	25	10					51.00
D06 Splay LG	0	20	60	7.0	1.0	10	10	5	11.8	30	20	10					19.80
D07 Griffon	0	20	60	20.0	10.0	10	15	5	21	45	25	10					51.00

Table 14-11: Variogram parameters used for the Northern Model Sulphur and NCC estimation.

SULPHUR VARIOGRAM PARAMETER TABLE

1201_NRM

AREA	Z Rotation	X Rotation	Y Rotation	Nugget	1st Rotation Spherical Sill	Range (y)	Range (x)	Range (z)	2nd Rotation Spherical Sill	Range (y)	Range (x)	Range (z)	3rd Rotation Spherical Sill	Range (y)	Range (x)	Range (z)	Total Varianc
	meds rotation																
D98 DOMA_ALL	0	20	50	0.19	0.295	12	10	5	0.75	33	20	10					1.24

Variogram parameters used for gold in the Phoenix Block Model (1512_PRM) estimation are listed in Table 14-12
Variogram parameters used for sulphur and NCC in the Phoenix Block Model (1512_PRM) estimation are listed in Table 14-13
Variogram parameters used for gold in the Central Block Model (1506_CRM) estimation are listed in Table 14-14
Variogram parameters used for sulphur and NCC in the Central Block Model (1506_CRM) estimation are listed in Table 14-15

Table14-12 VariogramparametersusedforthePhoenixModel (1512_PRM) Gold Estimation.

1512_PRM(PhoenixModel)	**GOLD VARIOGRAM PARAMETER TABLE**													
AREA	Z Rotation	X Rotation	Y Rotation	Nugget	1st Rotation Spherical Sill	Range (y)	Range (x)	Range (z)	Z Rotation	X Rotation	Y Rotation	2nd Rotation Spherical Sill	Range (y)	Ran (x
	meds rotation								*meds rotation*					
D01Audax	7	47	-10	0.62	0.66	60	45	20						
D01Audaxvar2	294	33	40	0.62	0.66	60	45	20						
D03PhoenixHG	10	30	50	20.00	10.00	10	15	5	10	30	50	21.00	45	2
D03Phoenix HGvar 2	260	-50	5	20.00	10.00	10	15	5	260	-50	5	21.00	45	2
D05SplayHG	0	20	70	7.00	1.00	10	10	5	0	20	70	11.80	30	2
D05Splay HGvar 2	60	46	30	7.00	1.00	10	10	5	60	46	30	11.80	30	2
D06Splays LG	0	20	70	7.00	1.00	10	10	5	0	20	70	11.80	30	2
D06Splays LGvar 2	260	-50	5	7.00	1.00	10	10	5	260	-50	5	11.80	30	2
D06 Splays LGoutlier estimation	5	15	70	7.00	1.00	9	9	9	5	15	70	11.80	9	9
D08Allwood	6	2	50	7.00	1.00	10	10	5	6	2	50	11.80	10	1
D08Allwood var 2	5	16	60	7.00	1.00	10	10	5	5	16	60	11.80	10	1
D08Allwood var 3	35	30	15	7.00	1.00	10	10	5	35	30	15	11.80	10	1
D09Vertical	1	10	90	0.61	0.34	70	50	25						
D09Verticalvar2	2	10	272	0.36	0.92	50	25	20	2	10	272	0.56	50	2
D09Verticalvar3	180	13	90	0.61	0.34	70	50	25						
D09Verticalvar4	6	25	-40	20.00	10.00	70	50	25	6	25	-40	21.00	70	5
D09Verticaloutlier estimation	5	15	35	20.00	10.00	9	9	9	5	15	35	21.00	9	9
D10BenuW1	18	23	207	0.32	1.06	20	10	10	18	23	207	0.74	20	1
D12PhoenixBase	57	35	5	20.00	10.00	10	15	5	57	35	5	21.00	45	2
D12PhoenixBasevar2	50	45	30	20.00	10.00	10	15	5	50	45	30	21.00	45	2
D12PhoenixBaseoutlierestimation	355	15	50	20.00	10.00	9	9	9	355	15	50	21.00	9	9
D13Benu	5	15	70	0.37	0.13	32	20	10	5	15	70	0.01	57	3
D13Benuvar2	285	-50	20	0.37	0.13	32	20	10	285	-50	20	0.01	57	3
D13Benuvar3	10	7	60	0.37	0.13	32	20	10	10	7	60	0.01	57	3
D13Benuoutlierestimation	5	15	70	0.37	0.13	7	7	7	5	15	70	0.01	7	7
D14BenuFW	5	10	35	0.73	0.55	60	30	15						
D14BenuFWvar2	75	50	12	0.73	0.57	60	30	15						
D14BenuFWoutlierestimation	5	10	35	0.73	0.57	9	9	9						

1512_PRM (Phoenix Model)	GOLDVARIOGRAMPARAMETERTABLE													
AREA	Z Rotation	X Rotation	Y Rotation	Nugget	1st Rotation Spherical Sill	Range (y)	Range (x)	Range (z)	Z Rotation	X Rotation	Y Rotation	2nd Rotation Spherical Sill	Range (y)	Ran (x
	meds rotation								*meds rotation*					
D15Kestrel	0	15	67	0.06	0.10	60	40	25						
D15Kestrelvar2	358	23	90	0.06	0.10	60	40	25						
D16BeddedEast	358	15	-80	0.06	0.10	70	50	25						
D16BeddedEastvar2	345	51	-75	0.06	0.10	70	50	25						
D16Bedded East outlier estimation	5	10	35	0.73	0.57	5	5	5						
D17ShallowEastDippers	8	17	-38	0.06	0.10	70	50	25						
D18EastDippers	358	15	-47	0.06	0.10	70	50	25						
D18East Dippers var 2	280	50	5	0.06	0.10	70	50	25						
D18EastDippersoutlierestimation	5	10	35	0.73	0.57	5	5	5						
D19Sphinx	**NotReportedduetoInsufficientNuggetDetermination**													
D20Eagle	0	35	305	0.49	1.26	70	50	25	0	35	305	0.76	70	5
D20Eagle	5	25	130	0.494	1.258	70	50	25	5	25	130	0.76	70	5
D20Eagle outlier estimation	5	10	35	0.727	0.574	6	6	6						

Table14-13 Variogram parameters used for the Phoenix Model (1512_PRM)SulphurandNCCestimation.

	SULPHUR/NCC VARIOGRAM PARAMETER TABLE													
1512_PRM(Phoenix Model)														
AREA	Z Rotation	X Rotation	Y Rotation	Nugget	1st Rotation Spherical Sill	Range (y)	Range (x)	Range (z)	Z Rotation	X Rotation	Y Rotation	2nd Rotation Spherical Sill	Range (y)	Rang (x)
	meds rotation								*meds rotation*					
D99WasteSulphur	50	20	50	0.30	0.19	12	10	5	50	20	50	0.30	33	20
D01AudaxSulphur	7	47	-10	0.30	0.19	8	10	8	7	47	-10	0.30	30	30
D03PhoenixHGSulphur	10	30	50	0.30	0.19	8	10	8	10	30	50	0.30	30	30
D05SplayHGSulphur	0	20	70	0.30	0.19	8	10	8	0	20	70	0.30	30	30
D06Splays LGSulphur	0	20	70	0.30	0.19	8	10	8	0	20	70	0.30	30	30
D08AllwoodSulphur	6	2	50	0.30	0.19	8	10	8	6	2	50	0.30	30	30
D09VerticalSulphur	1	10	90	0.30	0.19	8	10	8	1	10	90	0.30	30	30
D12PhoenixBaseSulphur	57	35	5	0.30	0.19	8	10	8	57	35	5	0.30	30	30
D13BenuSulphur	5	15	70	0.30	0.19	8	10	8	5	15	70	0.30	30	30
D14BenuFWSulphur	5	10	35	0.30	0.19	8	10	8	5	10	35	0.30	30	30
D15KestrelSulphur	0	15	67	0.30	0.19	8	10	8	0	15	67	0.30	30	30
D16BeddedEastSulphur	358	15	-80	0.30	0.19	8	10	8	358	15	-80	0.30	30	30
D17ShallowEastDippersSulphur	8	17	-38	0.30	0.19	8	10	8	8	17	-38	0.30	30	30
D18EastDippersSulphur	358	15	-47	0.30	0.19	8	10	8	358	15	-47	0.30	30	30
D19SphinxSulphur	5	36	70	0.30	0.19	8	10	8	5	36	70	0.30	30	30
D20EagleSulphur	0	35	305	0.30	0.19	8	10	8	0	35	305	0.30	30	30
D90NCCOD	0	15	-65	0.01	0.01	20	10	5	0	15	-65	0.00	50	30
D91NCCOD	0	15	75	0.01	0.01	20	10	5	0	15	75	0.00	150	75

Table14-14 Variogramparameters used for the Central Model (1506_CRM)Gold estimation.

1506_CRM(Central Model)	**GOLD VARIOGRAM PARAMETER TABLE**														
AREA	Z Rotation	X Rotation	Y Rotation	Nugget	1st Rotation Spherical Sill	Range (y)	Range (x)	Range (z)	Z Rotation	X Rotation	Y Rotation	2nd Rotation Spherical Sill	Range (y)	Range (x)	Ra
	meds rotation								*meds rotation*						
D01FostervilleHG	116	65	-50	3.70	5.70	7	5	5	116	65	-50	4.8	48	55	
~~D01FostervilleHG outlierestimation~~	355	15	50	20.00	10.00	7	7	7							
D02FostervilleHG	0	20	70	3.70	5.70	7	5	5	0	20	70	4.8	48	55	
D03PhoenixHG	10	30	50	20.00	10.00	10	15	5	10	30	50	21.0	45	25	
D03Phoenix HGvar 2	260	-50	5	20.00	10.00	10	15	5	260	-50	5	21.0	45	25	
D04PhoenixLG	0	20	50	2.40	1.00	10	15	5	0	20	50	2.65	35	25	
D05SplayHG	0	20	70	7.00	1.00	10	10	5	0	20	70	11.8	30	20	
D05Splay HGvar 2	60	46	30	7.00	1.00	10	10	5	60	46	30	11.8	30	20	
D06Splay LG	0	20	70	7.00	1.00	10	10	5	0	20	70	11.8	30	20	
D06Splay LGvar 2	260	-50	5	7.00	1.00	10	10	5	260	-50	5	11.8	30	20	
D07Kite	5	25	50	7.00	1.00	10	10	5	5	25	50	11.8	30	20	
D07Kite var 2	270	-45	5	7.00	1.00	10	10	5	270	-45	5	11.8	30	20	
D10Vulture	10	20	50	2.00	2.00	25	20	5	10	20	50	2.5	60	35	
D10Vulture var 2	91	50	-10	2.00	2.00	25	20	5	91	50	-10	2.5	60	35	
D10Vultureoutlierestimation	355	15	50	20.00	10.00	7	7	7							
D11HarrierOP	350	0	75	2.52	2.02	10	5	5	350	0	75	2.9	28	30	
D11HarrierOPvar2	55	70	30	2.52	2.02	10	5	5	55	70	30	2.9	28	30	
D12PhoenixBase	57	35	5	20.00	10.00	10	15	5	57	35	5	21.0	45	25	
D12PhoenixBasevar2	50	45	30	20.00	10.00	10	15	5	50	45	30	21.0	45	25	
~~D12Phoenix Baseoutlierestimation~~	355	15	50	20.00	10.00	7	7	7							
D18EastDippers	358	15	-47	0.06	0.10	125	50	25							
D18East Dippers var 2	338	53	-58	0.06	0.10	125	50	25							

Table14-15 Variogramparameters used for the Central Model (1506_CRM)Sulphur andNCC estimation.

SULPHUR/NCCVARIOGRAMPARAMETERTABLE

1506_CRM(Central Model)

AREA	Z Rotation	X Rotation	Y Rotation	Nugget	1st Rotation Spherical Sill	Range (y)	Range (x)	Range (z)	Z Rotation	X Rotation	Y Rotation	2nd Rotation Spherical Sill	Range (y)	Range (x)
	meds rotation								*meds rotation*					
D99WasteSulphur	0	20	50	0.30	0.19	12	10	5	0	20	50	0.30	33	20
D98DomaallSulphur	0	20	70	0.30	0.19	8	10	8	0	20	70	0.30	30	30
D98Domaall Sulphur var 2	35	60	60	0.30	0.19	8	10	8	35	60	60	0.30	30	30
D90NCCOD	0	15	-65	0.01	0.01	20	10	5	0	15	-65	0.02	150	75
D91NCCOD	0	15	70	0.01	0.01	20	10	5	0	15	70	0.02	150	75
D92(Waste)NCCOD	0	15	-65	0.01	0.01	20	10	5	0	15	-65	0.02	150	75

14.1.5 Resource Modeling

Block Models

For reasons of data handling, the Central Area was divided into three separate block models – Northern Central and Phoenix with the following extents and block dimensions contained within (Table 14-16)(Figure 14-22):

Table 14-16 Central Area Block Model dimensions.

Parameter	Northern	Central	Phoenix
Northing Min (m)	8,250	6,000	6,000
Northing Max (m)	10,250	8,250	8,250
Easting Min (m)	1,400	1,400	1,400
Easting Max (m)	2,100	2,100	1,850
RL Max (m)	5,200	5,200	4,600
RL Min (m)	4,800	4,600	4,000
X direction m (East)	2	2	2
Y direction m (North)	10	10	10
Z direction m (Vertical)	5	5	5

All models use Ordinary Kriging to interpolate grades.

Top Cuts

Historically, gold grades generated by disseminated sulphides were top cut to 75 g/t Au to limit the influence of a low number of high-grade intercepts. This changed in late 2014 and on into 2015 where the Central Area saw an increase in the number of high-grade assays associated with visible gold intersections into the Phoenix, Lower Phoenix and Eagle areas. The sudden change of support (Table 14-17) initiated a review of applicable top cuts and sourcing independent advice from QG consultants.

Table 14-17 Comparison between number of composites present above the cut off value between 2014 and 2015 for the same resource area.

Model Year	2m Composite Grade Cut-off (g/t Au)					
	25 g/t	50 g/t	75 g/t	100 g/t	150 g/t	200 g/t
	Number of Composites above Grade Cut-off					
2014	72	20	14	10	8	5
2015	208	75	60	46	34	23

On review of the Central Area dataset, QG recommended utilizing the Coefficient of Variation (CV) statistic as an outlier identification tool. The methodology suggested involved sorting the sample population to be analyzed from minimum to maximum grade and then the CV is calculated iteratively by removing the highest grades sequentially from the data set, The calculated CV's should be plotted against the number of samples removed from the population. This produces a curve that starts with a steep negative slope and gradually flattens to approach the horizontal. The rate of change in the CV will decrease as more and more samples are removed from the dataset. A reasonable indication of outlier grade can be determined by examining the rate of change in CV and selecting a point where the impact of removing another sample is relatively low.

As with most statistical analysis methods, the approach is reliant on the number of data points within the population. The relative ease of generating the dataset and plots allows consistent review and updating of applicable top cuts from model to model.

With review of the below graphs, the D09 Vertical (Figure 14-18) and D10 Benu W1 (Figure 14-19) domains show top cuts that are not aligned with their plotted CV changes. The D10 Benu W1 suffers from the sample support being a high grade but narrow domain. The D09 Vertical domain is well supported; however, caution has been applied to directly applying the upper CV value top cut due current disparities with mill reconciled values. The D13 Benu (Figure 14-20) domain has a significant inflection on the CV trend line at between 56 and 88 g/t. This is in line with the top cut sought for the interpolation at 75g/t. The D20 Eagle (Figure 14-21) domain data exhibits a CV level of just above 1.0 at the 93g/t level. Again this is consistent with the top cut being optimized at 90g/t.

Using the methodology, the Phoenix Resource Model had a number of component domains have their top cuts increased (Table 14-18).

Importantly, model vs mill reconciliation performance is reviewed in parallel with the statistical analysis of data populations when deciding on top cut application. As data populations increase through additional drilling and mining in the visible gold environments of Lower Phoenix top cuts will be revised on an ongoing basis.

In addition to the Au top cut, an interpolation range limit was imposed on samples in most of the domains within the Phoenix 1512_PRM Resource model. The range limits imposed, vary between 10 and 60m for assay grades varying between 12-30 g/t Au. This limit was imposed to decrease the importance of locally estimated blocks with very high composite sample grades (>70 g/t Au) where sample support was low.

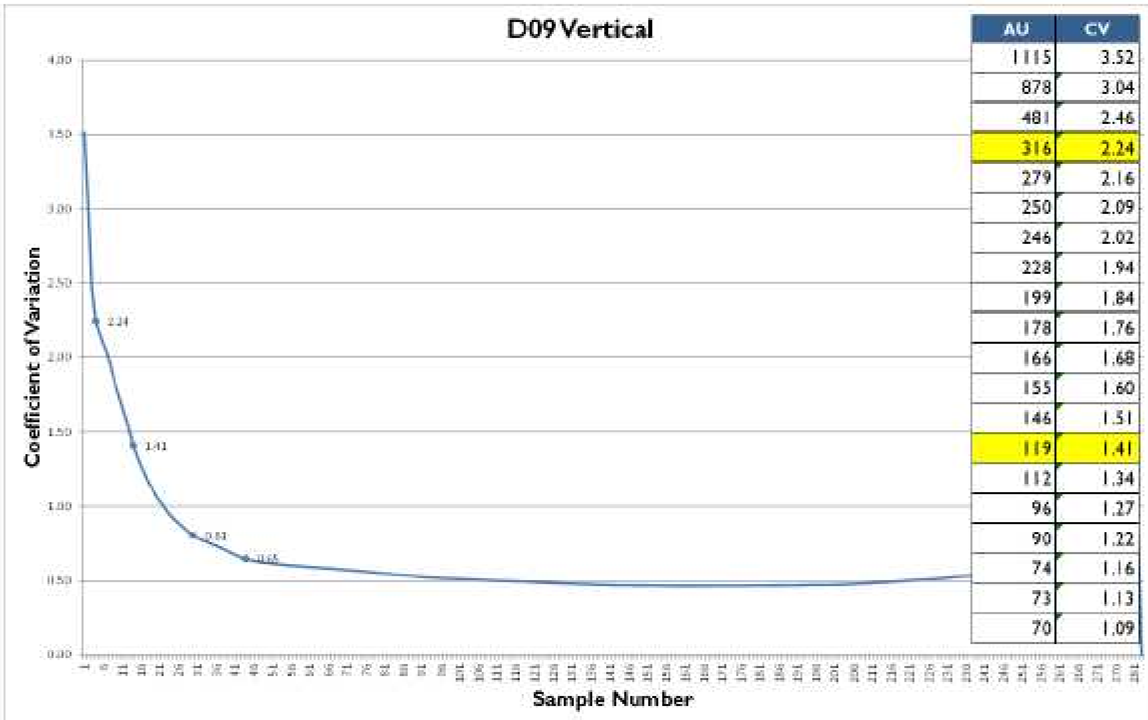

Figure 14-18 Coefficient of Variation method applied to the D09 Vertical Domain

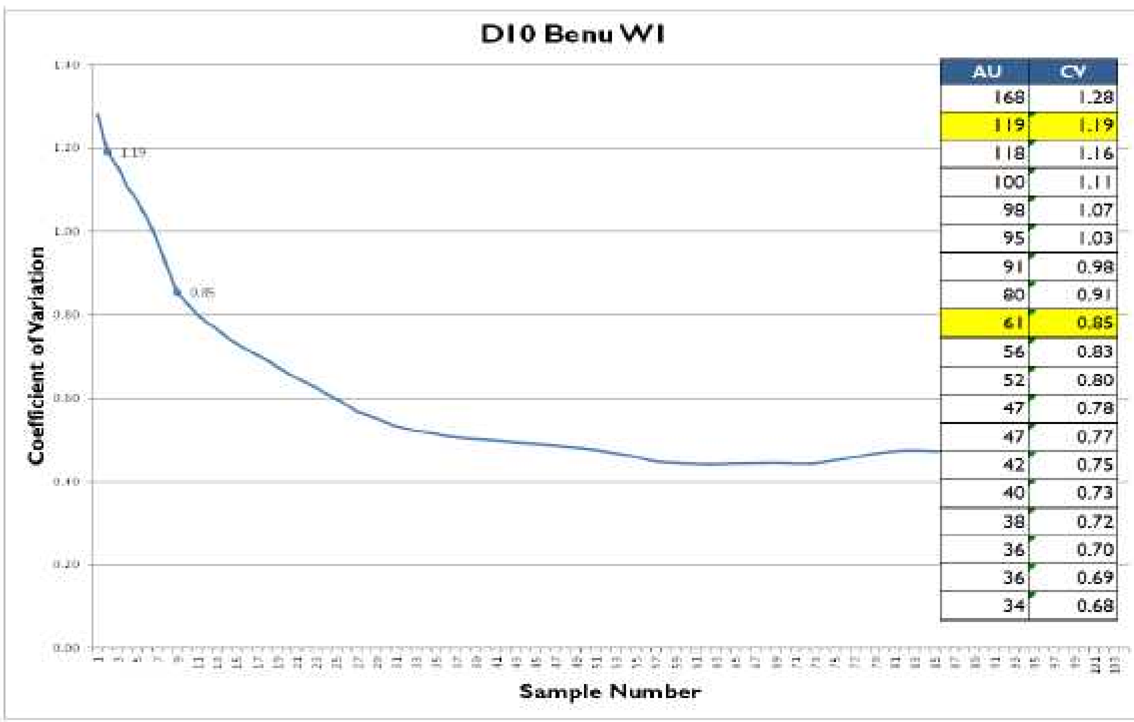

Figure 14-19 Coefficient of Variation method applied to the D10 Benu W1 Domain

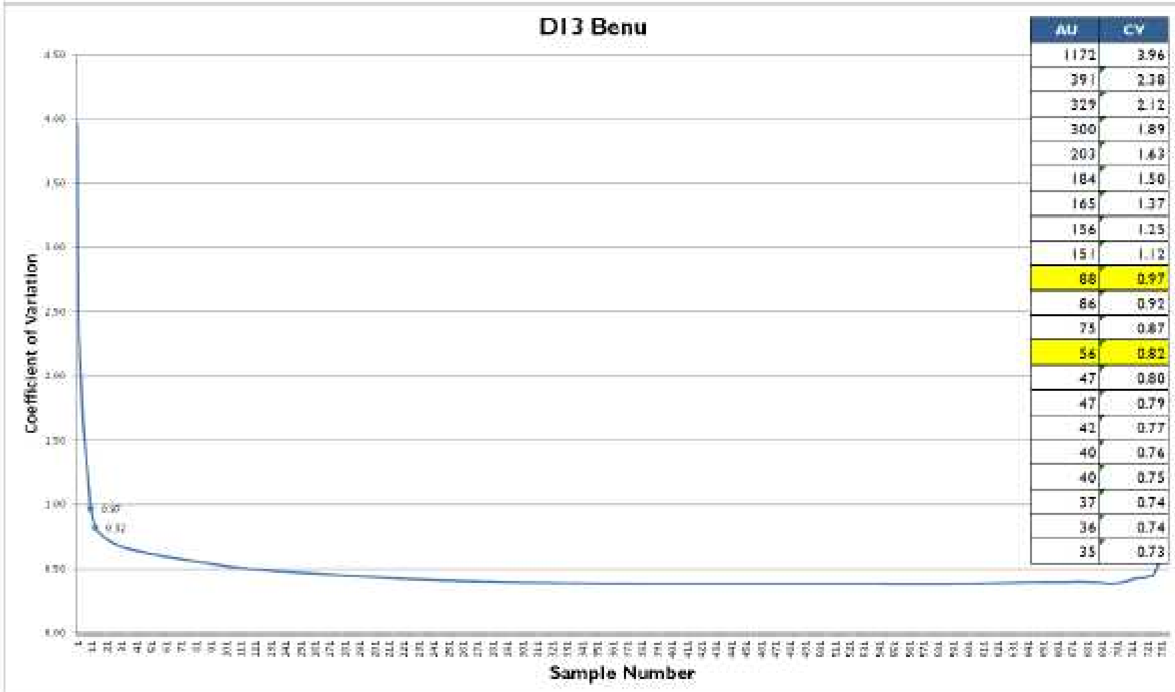

Figure 14-20 Coefficient of Variation method applied to the D13 Benu Domain

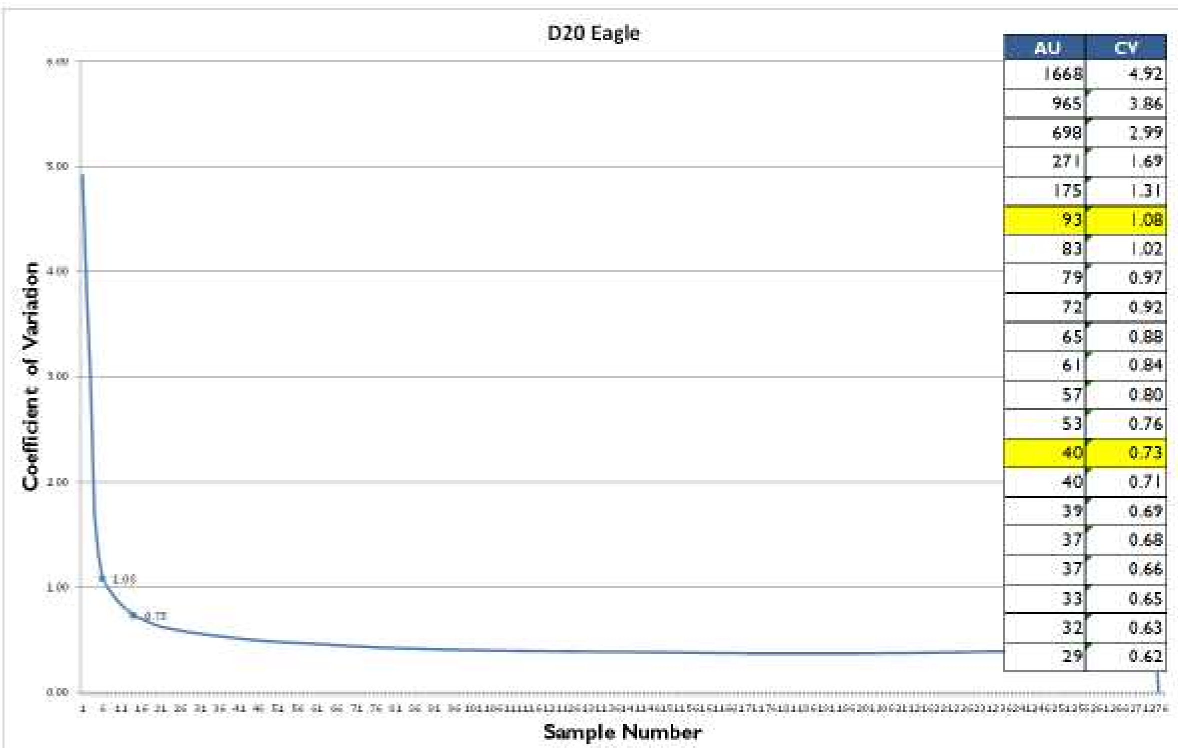

Figure 14-21 Coefficient of Variation method applied to the D20 Eagle Domain

Table 14-18 Top cuts applied within the Phoenix Resource Model 2015, g/t Au

Domain Name	Domain Number	Top Cut Applied
Audax	1	75
Phoenix HG	3	75
Splay HG	5	75
Splay LG	6	75
Allwood	8	75
Vertical	9	**120**
Benu W1	7	**100**
Phoenix Base	8	75
Benu	9	75
Benu FW	14	75
Kestrel	15	75
Bedded East	16	75
Shallow East Dippers	17	75
East Dippers	18	75
Sphinx	19	75
Eagle	20	**90**

Search Criteria

Gold, Sulphur and NCC grades are only interpolated into blocks meeting the following criteria:

Greater than 1% of the block volume is inside one of the domain envelopes

Blocks within one of the Resource domain category solids

Blocks whose search ellipse includes at least 1 composite, depending on the particular mineralized envelope

The search ellipse geometries were chosen to reflect the geology, variogram model and drill spacing of the relevant zone so that a block could 'see' at least the nearest sections along strike and holes up or down dip ..

Only composites meeting the following criteria are used to interpolate any one block, where:

Composites (to a maximum of 35) within the search ellipse dimensions and search area limits.

More than 35 composites lie within the search ellipse, the closest 35 samples in anisotropic ellipsoid space are used.

There was no directional de-clustering employed in the 1512_CRM Model or the 1506_CRM Model. A maximum of 10 composites per quadrant were estimated in a four sector quadrant search in the 1201_NRM Model (Table 14-21).

Codes of both the composite and the block were matched by correlating the coded composite item with the coded block model item.

A maximum of four composites can be taken from any single drill hole.

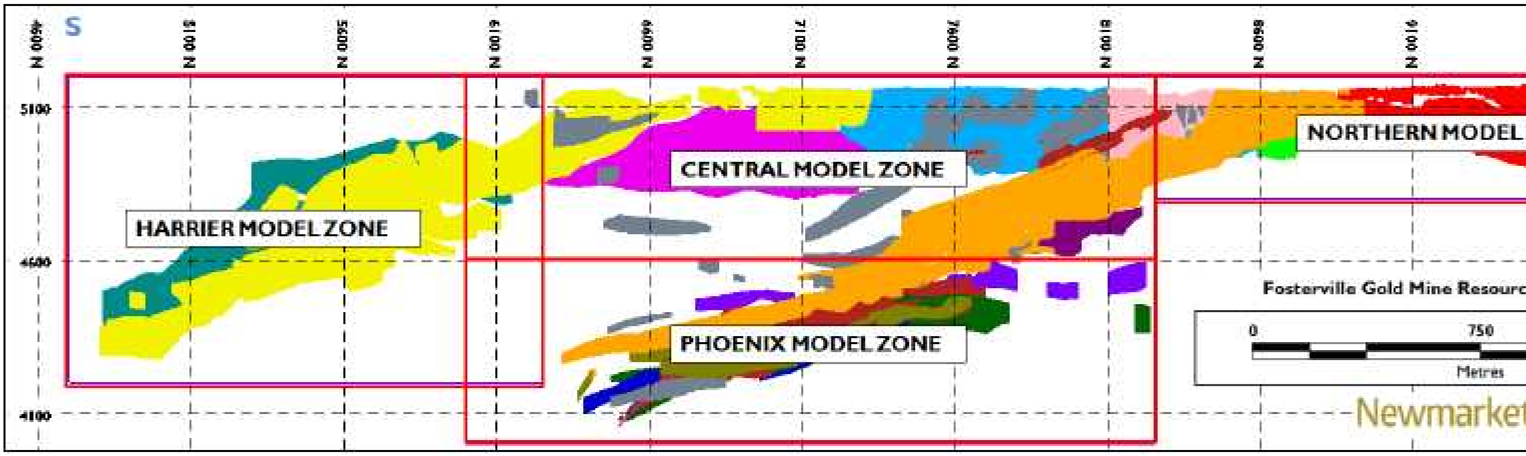

Figure 14-22 Long Projection showing Northern, Central and Harrier Model extents as of December 31, 2015

To ensure the suitability of the search ellipses used for interpolation, a Kriged 'de-bug' search ellipse was created in MineSight on selected domains for the variogram to be used allowing visual inspection of the composites and Kriging weights calculated for the block at the center of the ellipse.

An additional 'outlier' sub domain was introduced within the 1512_PRM Model for the Phoenix, East Dippers and Eagle Zones. Similarly, the 1506_CRM Model also had outlier sub-domains employed in selected domains. This was to restrict the influence of lower grade composite Au values smearing into high-grade areas of the resource shown in Table 14-20.

Search ellipses in Figure 14-23 show the maximum range extents that composites can be used to estimate a block. Range extents for the 1512_PRM Model can be seen in Table 14-19. Search routines used to interpolate blocks in the model are a combination of a broad extensive search based on a low sample support estimate combined with an overprint of a tighter higher supported estimation, particularly relevant in areas of tighter drill spacing.

The steep search ellipse criteria, limits the strike influence of the high-grade shoots described in Section 14.1.3. There is some geological evidence to suggest that the steep mineralized shoot geometry is a result of the intersection of splay faults with the main west dipping fault structures, however subtle changes in fault strike may also be responsible for this observed trend. This shoot geometry is usually associated with wider zones of mineralization increasing the number of composites available at a higher grade than is seen towards the edges of the mineralization. Support for a steeper fault intersection plunging shoot can be seen in the 1512_PRM model, specifically the DOMAIN=1 Audax and DOMAIN=9 Vertical intersection where the associated intersection plunge is steep to the south and is reflected in the second search included within the higher supported interpolation for the DOMAIN=1 Audax.

Similarly, the 1506_CRM Central Model includes a second intersection plunge shoot interpolation on a number of domains to capture both the extensive block interpolation on the periphery of the respective domain and to provide a higher level of block/sample support in those areas of tighter drill spacing. Range extents for the 1506_CRM Model can be seen inTable 14-20.

Figure 14-23 depicts the DOMAIN=13 Benu Resource with its first and second search ellipses for Gold interpolation. Rotations are in MineSight coordinates.

Table14-19 Search parameters for the Phoenix Model (1512_PRM)

1512_PRM(Phoenix model)	**SEARCHPARAMETERTABLE**														
DOMAIN	y axis	x axis	z axis	Rotation (z)	Rotation (x)	Rotation (y)	min. samples 1st search	max. samples 1st search	yaxis	x axis	z axis	Rotation (z)	Rotation (x)	Rotation (y)	min. samples 2nd search
	1st/3rd search distance			rotation			sample number definition		2nd/ 4thSearchDistance			rotation			sa num defin
D01Audax	90	50	20	7	47	-10	2	35	70	30	10	294	33	40	3
D03PhoenixHG	70	50	40	10	30	50	2	35	50	20	20	260	-50	5	4
D05SplayHG	80	60	40	0	20	70	1	35	50	40	20	60	46	30	4
D06Splays LG	115	105	50	0	20	70	2	35	50	20	20	260	-50	5	4
D06 Splays LGoutlier estimation	9	9	9	5	15	70	1	35							
D08Allwood	300	150	50	6	2	50	1	35	60	60	30	5	16	60	1
D08Allwood third/fourth search	50	25	15	35	30	15	4	35							
D09Vertical	180	60	20	1	10	90	1	35	50	25	20	2	10	272	2
D09Verticalthird/fourthsearch	60	30	20	180	13	90	3	35	60	40	20	6	25	-40	6
D09Verticaloutlierestimation	9	9	9	5	15	35	1	35							
D10BenuW1	40	30	15	18	23	207	2	35	20	15	10	18	23	207	3
D12PhoenixBase	80	50	30	57	35	5	1	35	60	30	30	50	45	30	5
D12PhoenixBaseoutlierestimation	9	9	9	355	15	50	1	35							
D13Benu	130	80	30	5	15	70	2	35	50	20	20	285	-50	20	2
D13Benuoutlier estimation	7	7	7	5	15	70	1	35							
D14BenuFW	100	60	30	5	10	35	2	35	70	30	20	75	50	12	5
D14BenuFWoutlierestimation	9	9	9	5	10	35	1	35							
D15Kestrel	100	80	40	0	15	67	1	35	70	50	20	358	23	90	3
D16BeddedEast	80	50	20	358	15	-80	1	35	40	20	10	345	51	-75	3
D16Bedded East outlier estimation	5	5	5	5	10	35	1	35							
D17ShallowEastDippers	90	55	30	8	17	-38	2	35							
D18EastDippers	110	80	30	358	15	-47	1	35	50	20	20	280	50	5	3
D18EastDippersoutlierestimation	5	5	5	5	10	35	1	35							
D19Sphinx	100	50	20	5	36	70	2	35	50	40	20	5	36	70	4
D20Eagle	65	40	30	0	35	305	2	35	35	15	5	5	25	130	5
D20Eagle outlier estimation	6	6	6	5	10	35	1	35							

Table14-20 SearchparametersfortheCentralModel (1506_CRM)

1506_CRM(Central model)	SEARCHPARAMETERTABLE														
DOMAIN	y axis	x axis	z axis	Rotation (z)	Rotation (x)	Rotation (y)	min. samples 1st search	max. samples 1st search	y axis	x axis	z axis	Rotation (z)	Rotation (x)	Rotation (y)	min. sample 2nd search
	1st/3rd Search distance			Rotation			Sample number definition		2nd/ 4thSearchDistance				Rotation		S nu defi
D01FostervilleHG	80	30	20	116	65	-50	2	35	80	20	20	116	65	-50	4
~~D01FostervilleHGoutlierestimation~~	7	7	7	355	15	50	1	35							
D02FostervilleHG	160	160	80	0	20	70	2	35	80	80	50	0	20	70	8
D03PhoenixHG	70	50	40	10	30	50	2	35	50	20	20	260	-50	5	4
D04PhoenixLG	85	75	40	0	20	50	2	35	70	40	40	0	20	50	8
D05SplayHG	80	55	45	0	20	70	1	35	50	40	20	60	46	30	4
D06Splay LG	110	90	40	0	20	70	2	35	50	20	20	260	-50	5	4
D07Kite	80	60	30	5	25	50	1	35	50	30	30	270	-45	5	4
D10Vulture	115	90	45	10	20	50	1	35	80	30	20	91	50	10	3
D10Vultureoutlierestimation	7	7	7	355	15	50	1	35							
D11HarrierOP	100	100	80	350	0	75	6	35	40	30	20	55	70	30	4
D12PhoenixBase	80	50	40	57	35	5	1	35	60	30	30	50	45	30	5
~~D12PhoenixBaseoutlierestimation~~	7	7	7	355	15	50	1	35	60	30	30	50	45	30	5
D18EastDippers	110	80	30	358	15	-47	1	35	80	40	20	338	53	-58	3

Table14-21 SearchparametersfortheNorthernModel(1201_NRM)

1201_NRM(Northern Model)	SEARCHPARAMETERTABLE																
DOMAIN	y axis	x axis	z axis	Rotation (z)	Rotation (x)	Rotation (y)	min. samples 1st search	max. samples 1st search	y axis	x axis	z axis	Rotation (z)	Rotation (x)	Rotation (y)	min. samples 2nd search	max. samples 2nd search	Qu sa sele
	1st/3rd search distance			rotation			sample number definition		2nd/ 4thSearchDistance				rotation		sample number definition		
D01FostervilleHG	100	100	50	0	20	70	2	35	40	40	20	0	20	70	5	35	
D02Fosterville LG	100	100	80	0	20	70	6	35	80	80	50	0	20	70	8	35	
D03PhoenixHG	120	120	50	355	20	50	3	35	60	50	30	355	20	50	8	35	
D06Splays LG	120	120	50	0	20	60	1	35	100	100	50	0	20	60	8	35	
D07Griffon	80	80	50	0	20	60	8										

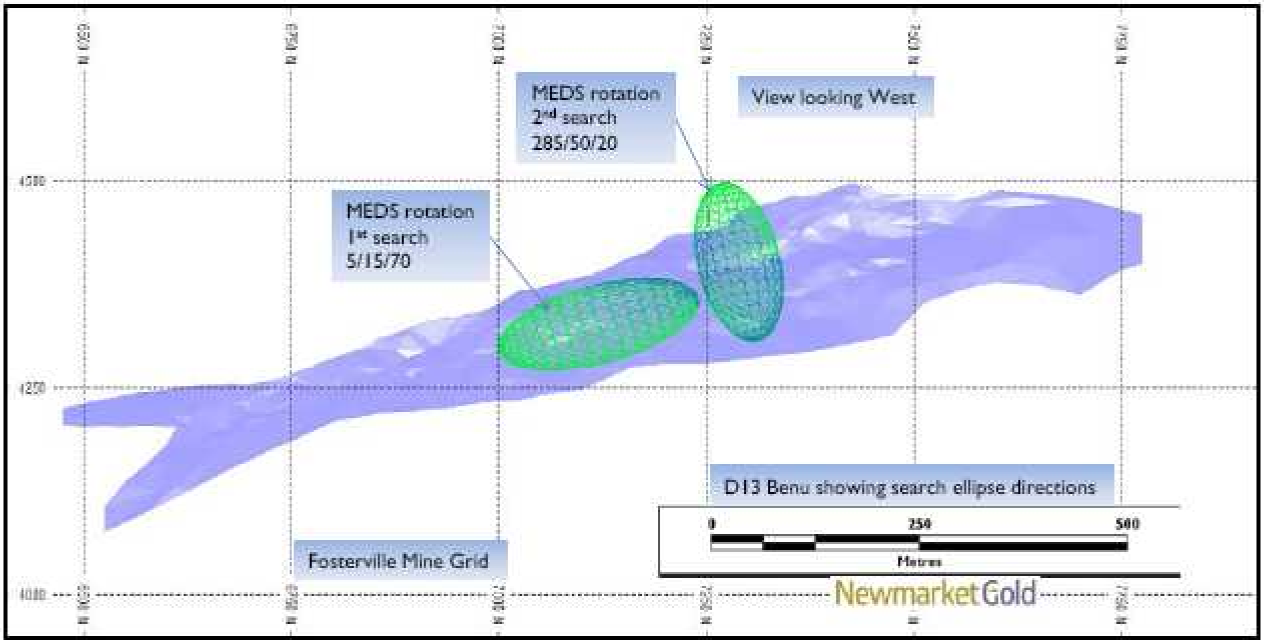

Figure 14-23 Search ellipses for DOMAIN=13 Benu

The resultant block models are therefore tightly constrained by wireframe models derived from detailed geological interpretation and modeling of the mineralized zones. This provides the vital basic geological control over the computer-generated grade estimations. A section through the block model is included in Figure 14-24.

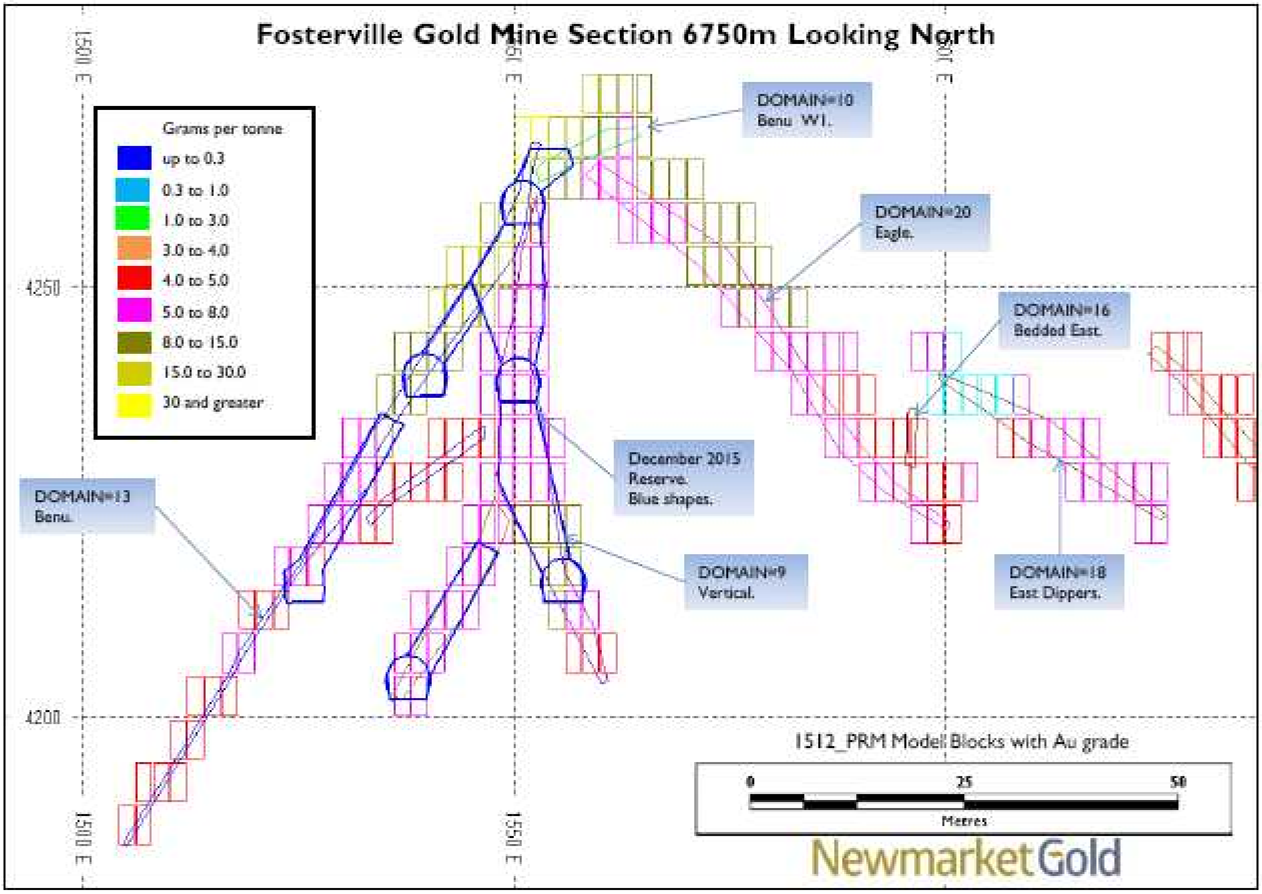

Figure 14-24 Section through the Central Model (6750mN), showing the DOMAIN=13 Benu, DOMAIN=9 Vertical, DOMAIN=10 Benu W1 and DOMAIN=20 Eagle mineralization envelope.

Bulk Density

During the course of 2013, a review was conducted of the bulk density values used at Fosterville including analysis of all diamond core data and grab sample analysis from known production locations. Bulk density measurements conducted on production samples via a water displacement method (Lipton, 1997) shows the average densities of mineralized material at 2.79 t/m^3, stibnite material at 3.20 t/m^3 and waste material at 2.76 t/m^3 (Table 14-22). Further support can be seen in Figure 14-25where a total of 1,078 samples of mineralized and un-mineralized samples were charted against their respective reduced level. From the graphs produced, it can be seen that data below 4500m RL are greater than the previously used model bulk density value of 2.72 t/m^3. When looking at only mineralized samples above a 1.0 g/t Au cut-off in Figure 14-26, there is a clear step change below 4500m RL with an average density of 2.80 t/m^3. It is important to note that data points around 4200m RL shown in Figure 14-26 show a drop in density, however, this is due to insufficient number of samples taken at around this level. A decision was made to increase model density below 4500m RL from 2.72 t/m^3 to 2.78 t/m^3 given the supporting evidence.

Table 14-22 Bulk density samples from underground production locations.

Source	Reduced Level (m)	Description	Calculated Density (t/m^3)
O4640	4640	Mineralized	2.77
O4640	4640	Mineralized	2.68
C4480	4480	Mineralized	2.75
C4480	4480	Mineralized	2.94
C4480	4480	Stibnite	3.52
C4460	4460	Mineralized	2.84
C4460	4460	Mineralized	2.75
C4460	4460	Stibnite	3.00
C4460	4460	Stibnite	3.07
C4460	4460	Waste	2.67
C4460	4460	Waste	2.77
C4480	4480	Waste	2.82
C4480	4480	Waste	2.79
O4640	4640	Waste	2.70
O4640	4640	Waste	2.79

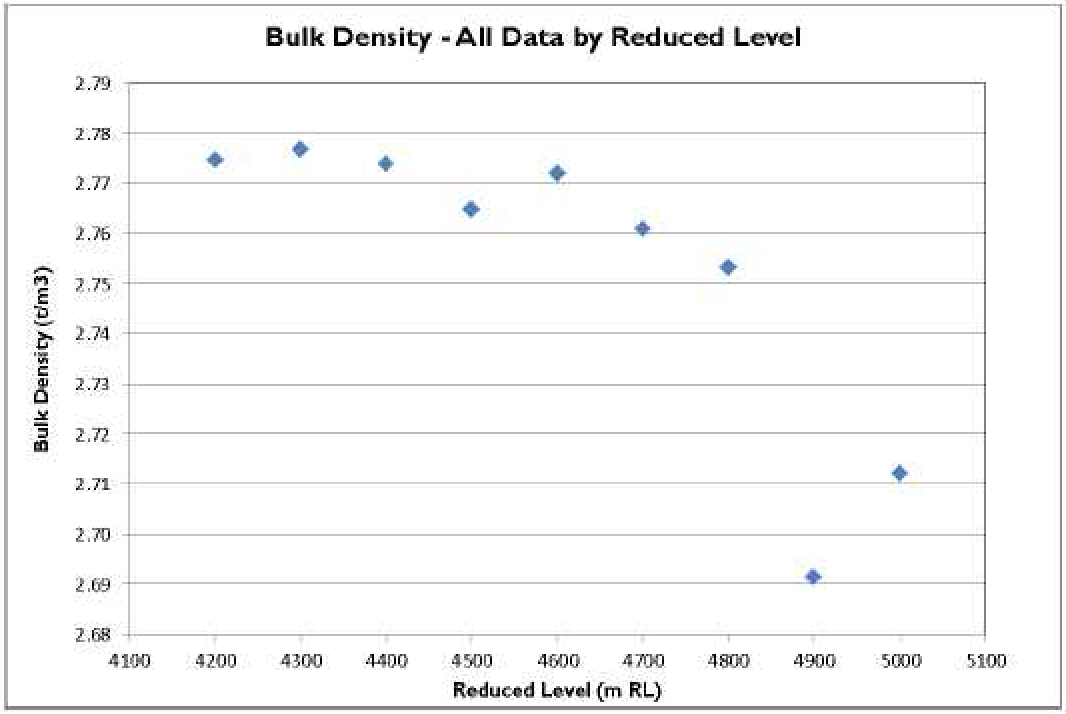

Figure 14-25 Diamond drill core bulk density values vs. reduced level for data up to October 2013.

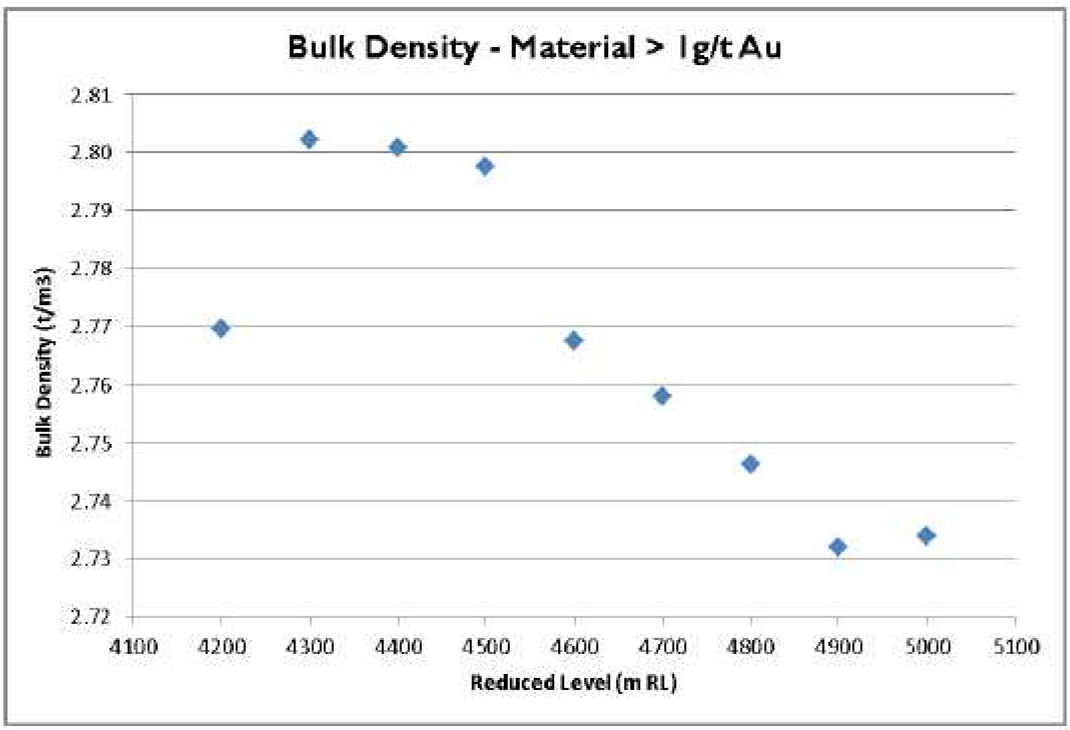

Figure 14-26 Diamond drill core bulk density values (intervals >1g/t Au) vs. RL for data up to October 2013.

Bulk Density within the oxide zone from surface to base of complete oxidation is determined from RC drilling, and test work assigns it a value of 2.40 t/m^3. Fresh rock is then divided into four zones determined by test work carried out on the diamond drill core. The three categories are based on relative level with transitional material between fresh and oxide above 5050mRL assigned 2.56 t/m^3, fresh material between 5050 and 5000mRL assigned 2.64 t/m^3, fresh material between 5000mRL and 4500mRL assigned 2.72 t/m^3 and fresh material below 4500mRL assigned 2.78 t/m^3 (Figure 14-27).

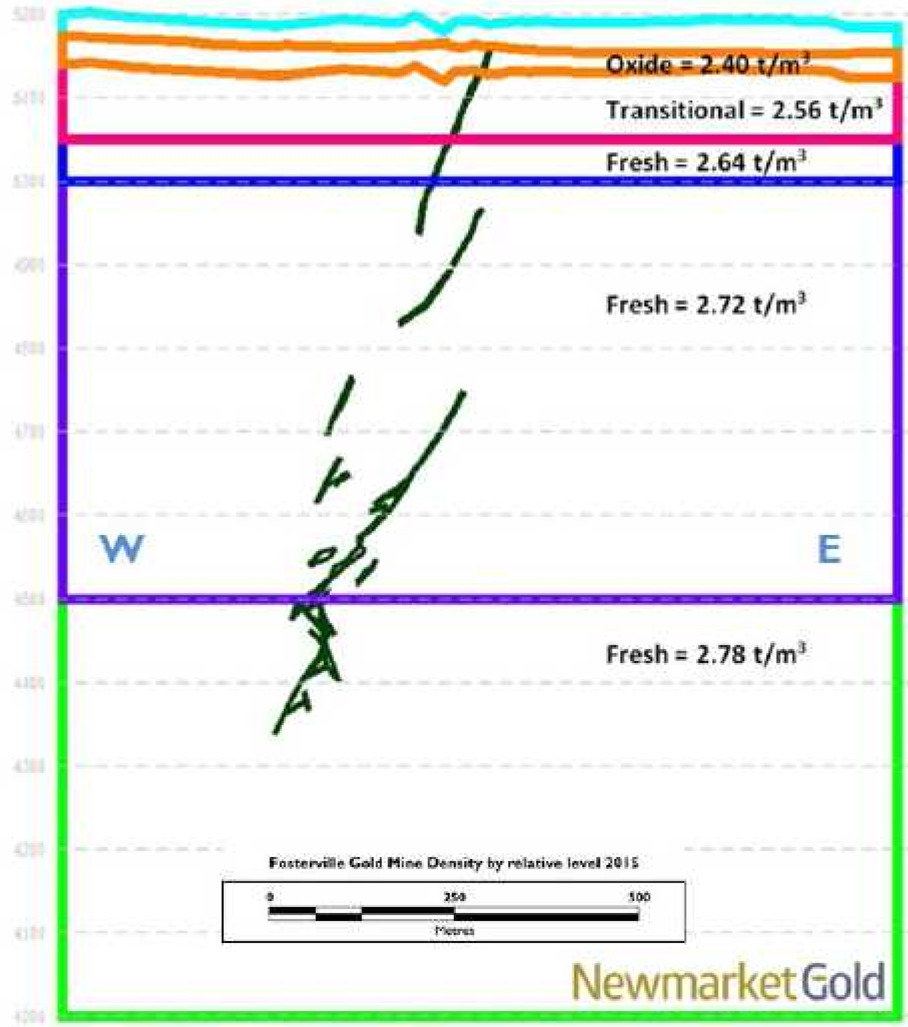

Figure 14-27 Bulk density values used in resource models.

Mineral Resource Classification

The mineral resource estimates were generally classified according to the following parameters:

- Areas that have proximal underground development (as a draw point to a stoping block) were classified as Measured Mineral Resources with the Resources having adjacent mapping, face sampling and sludge sampling through the area. This does not extend to the material in stoping blocks below the lowest developed level in the area.
- Areas drilled up to a spacing of 25m x 25m were classified as Indicated Mineral Resources.
- Areas drilled to spacing wider than 50m x 50m were classified as Inferred Mineral Resources.

These parameters may vary subject to the level of geological confidence in specific areas.

Figure 14-28 depicts Mineral Resource classifications and Reserves encompassing the Central and Phoenix Areas as at December 31, 2015.

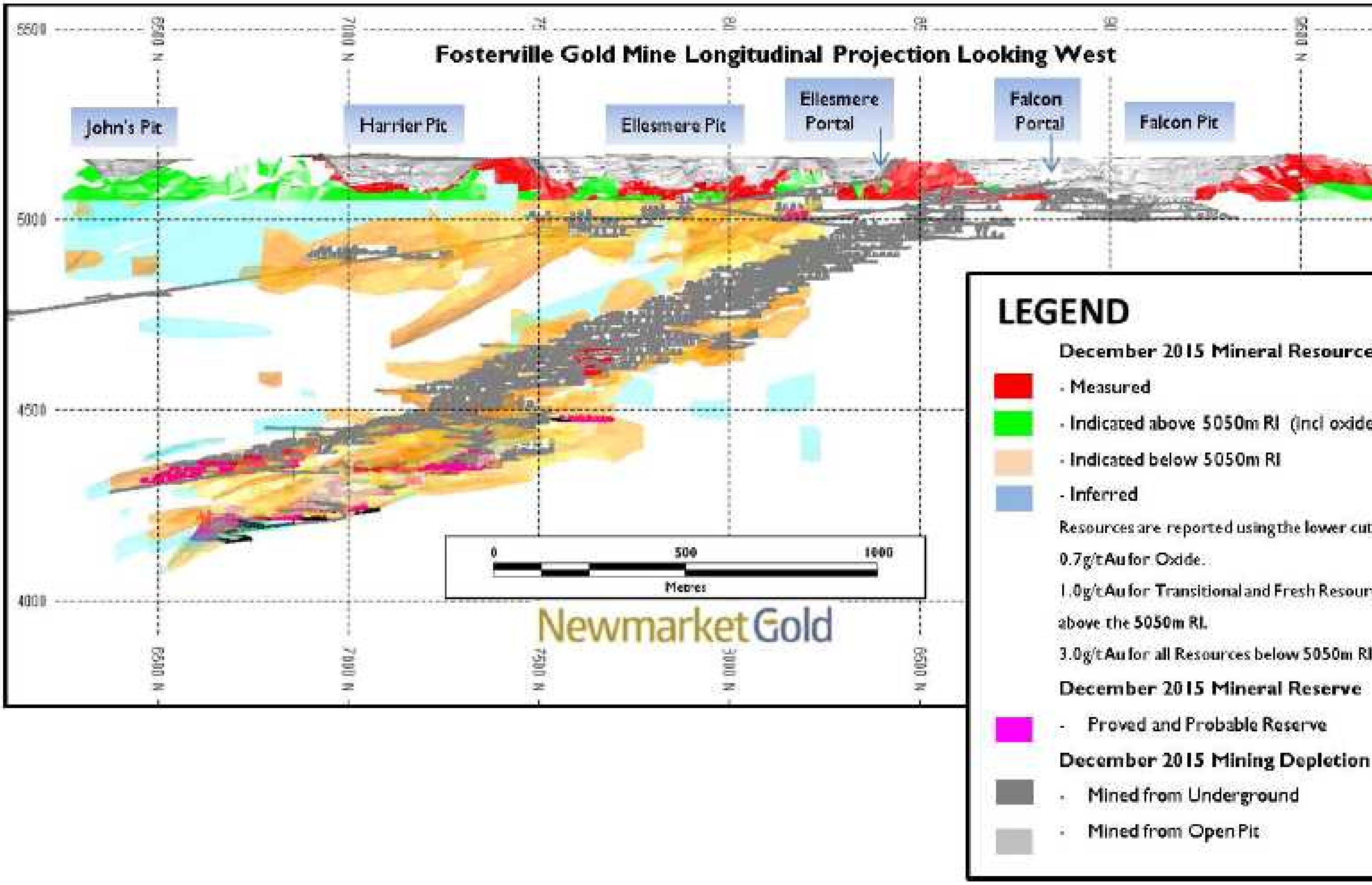

Figure 14-28 Longitudinal projection looking west, showing Mineral Resource classification.

14.1.6 Results

Results for the Mineral Resources contained in the Central Area (Central, Phoenix and Northern Model) are provided in Table 14-1.

14.2 Harrier Area

The Harrier UG Area resides within the bounds of the Southern Model Area and replaced the Wirrawilla region in 2009, while not encompassing the Daley's Hill Open Pit region. Project definitions and model boundaries were altered to coincide with the transition of the Harrier UG project from Exploration to Mine Geology (Figure 14-1).

In late 2009 a focused detailed review of the information gathered was undertaken to determine mining risk. Analogues derived from systems developed to understand Central Area geology were applied to the Harrier UG dataset. While fundamental Fosterville geological principles such as the larger faulting systems, stratigraphy and plunge were found to be sound, the inter-relationship between structure and grade required further investigation.

14.2.1 Area Geology

Within the Harrier UG Model area, there appear to be two main zones of mineralization, one zone associated with the Harrier Fault System and the other with the Osprey Fault System. Both systems trace their roots back to movement along the Fosterville Fault; however appear to differ at their nucleation points with the Osprey System sitting higher in the system with relation to the Harrier System (Figure 14-29).

Both systems generate most of their fault related mineralization within oblique/oblique environments as movement propagates away from the Fosterville Fault. The systems are related by the way of linking structures that strike ~5° to the north as opposed to the Osprey and Harrier Systems that strike ~350° to the north. The relationship between structures takes on large en echelon type geometry with mineralization intensity increasing at the intersections between main systems and linking structures.

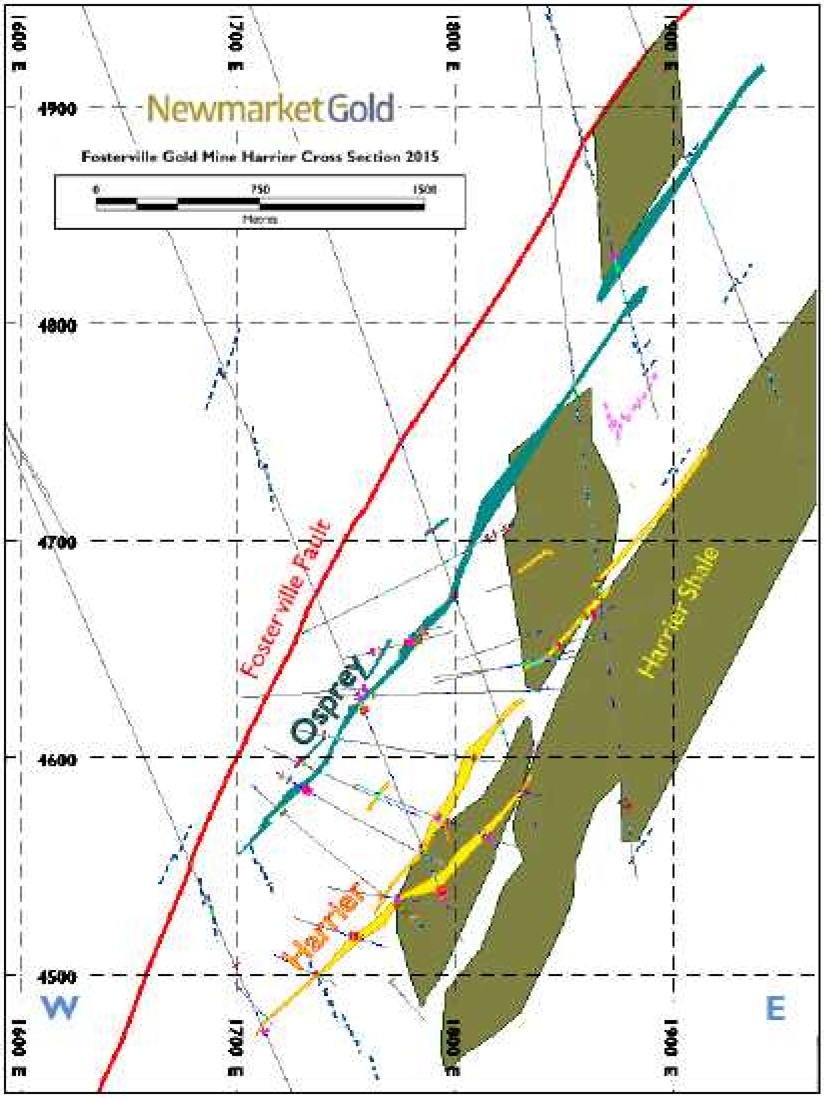

Figure 14-29 Geological cross section through the Harrier Area at 5312.5mN.

Initial drilling was conducted from surface to build the Harrier Ore body on a 50m x 50m drill centered program. Since 2011, with the progression of the Harrier decline, drill centers have been drawn into a 25m x 25m drill spacing allowing domains to be built into 25m sections. Areas of particular geological difficulty have also be drilled into 12.5m centers with domains also being constrained using underground face mapping, sampling and sludge hole sampling data. Drill program progress was improved with the addition of the Harrier 4625RL Diamond Drill Drive which provided resource definition as far south as 4850mN. In 2016 there is a planned drilling program of approximately 32,208 meters. The aim is to improve the drilling spacing to 25m x 25m southwards to approximately the 4900mN to support mining studies.

14.2.1 Geological Modeling

Geology Models were developed using similar methodology to that discussed in 14.1.2 within the Central Area

14.2.2 Domains

Based on observed variations in geology, variography, geochemistry, statistics and spatial location within the Fosterville mine area, mineralization in the Harrier Area has been divided into nine unique domains and one common Splay domain shared with the Central Area. The domains and domain codes corresponded to:

6. Splay LG (Low-grade) - *common to Harrier and Central Areas*
20. Harrier
21. Harrier Base
22. Harrier Link
24. Harrier HW - *HW= Hanging wall*
25. Harrier Splay
30. Osprey
31. Osprey Base
32. Osprey Link
35. Osprey Splays

The domains can be generically categorized into two groups, Harrier (5 Domains) Figure 14-30, Osprey (4 Domains) Figure 14-31 including various Splays. Harrier and Osprey domain differentiations are driven primarily on grade population differences between structures that reside within close proximity to each other. The host geology of the mineralization within the Harrier UG Area is consistent with details listed within the Central Area.

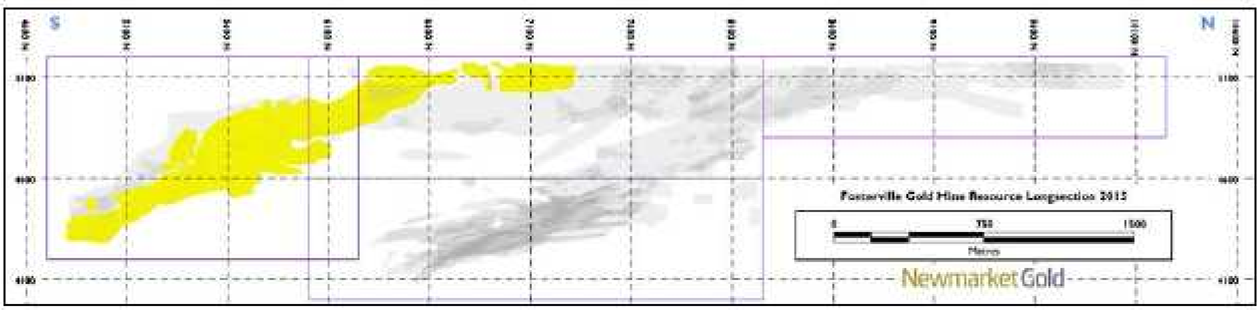

Figure 14-30 Longitudinal projection of Harrier Mineralized Zone

The Harrier System is interpreted to have developed as reverse thrust faulting progressed up the Fosterville Fault reaching the anticline, refracting and developing a complex system of splay faults that link across to the eastern syncline hinge. Fault propagation continues across east dipping interbedded sandstone and shale beds before movement conforms into the large Harrier Shale package. Movement into the Eastern syncline and Harrier Shale package develops several minor hinge offsets along early LQ veins that create localized zones of oblique/parallel mineralization.

The Harrier Shale package proximal to the orebody has been is estimated to be ~30m in thickness with several LQ veins throughout the succession. Major LQs were correlated along strike and structurally wireframed to create the Harrier Base and Harrier Upper Faults. The total displacement over the Harrier suite of faults is about 120m.

The Harrier Mineralized Zone extends through to surface having been mined as the Harrier Open Pit with its northern most extent around 7300mN. The system has an overall plunge of 25^{o} with the main underground shoot of mineralization not beginning until around the 4760mRL. The Harrier Zone consists of 5 distinct domains including the Harrier, Harrier Base Harrier Link, Harrier Splay and Splay LG Domains.

Mineralization within the Harrier Zone consists of primary sulphides including arsenopyrite and pyrite with the area having only localized amounts of stibnite. The sulphides are disseminated into the host sandstone and shale packages around strongly faulted and fractured areas. Grade tenor proxies utilized in the Central Zone such as the percentage of arsenopyrite can be misleading due to mica rich sand horizons being mistaken for mineralization, silicification of host rocks giving a false indication of quartz fluid flow and fine sulphide crystal growth that can be overlooked as dust or sedimentary fine grains.

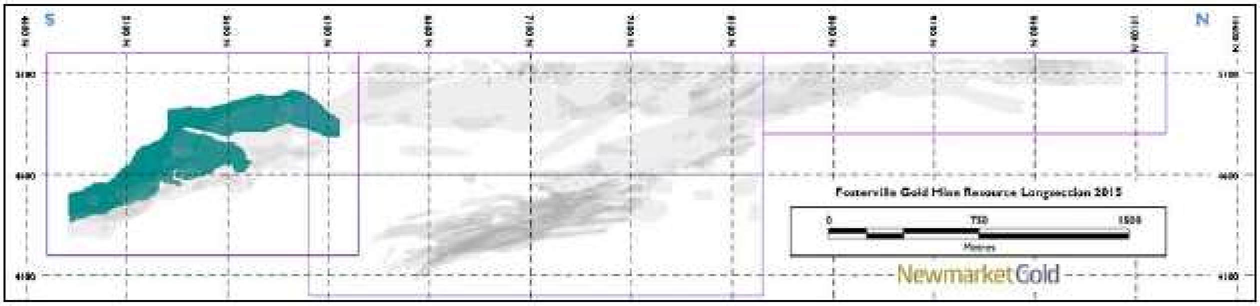

Figure 14-31 Longitudinal projection of Osprey Mineralized Zone

The Osprey System is modeled ~50m hangingwall to the Harrier system and appears to be the last splay fault that bifurcates from the Fosterville Fault before the Fosterville anticline. The movement seen on the Osprey appears to maintain its offset to the Harrier System up dip however does not appear to connect through to the eastern syncline hinge as the Harrier System does. There is growing support to suggest that mineralization in the Osprey System is directly influenced by the western limb of the Harrier Shale package as areas of intersection appear to act as a barrier to the flow of mineralization further up dip of the Osprey System.

The Osprey System shares similar geometries to that seen in the Harrier System with economic mineralization largely running in parallel between 5420mN and 5100mN. North of 5420mN, the Osprey system mineralization links across to the Harrier system utilizing the linking structures. South of 5100, the Osprey System appears to trend more north-south with similar trends to the second order linking structures. Structures that trend more north-south appear to take on a lower grade tenor than those that strike towards ~355°, although the controls on why this occurs is poorly understood and further drilling to the south is required to see if there is a data support issue with the interpretation

The Osprey System consists of four distinct geometries including the Osprey, Osprey Base, Osprey Link and Osprey Splays. The main shoot of Osprey mineralization is encompassed within the Osprey domain that is modeled south of 5725mN and remains open at depth. The Osprey System has similar geological properties to the Harrier System (Strike ~355°, Dip ~40°, and Plunge ~ 20°), however, gains some complexity around 5450mN where multiple converging geometries are modeled.

14.2.3 Drilling Data

Compositing

Similar to the Phoenix Model (1512_PRM), coded Harrier drill data was composited to 2m lengths with a 1m add-back threshold to avoid an increase in small intervals close to the margins of the coded mineralized domains. If the final composite was less than 1m, it was added to the previous composite making a composite with a length between 2m and 3m, refer to Figure 14-17**.** Final composites between 1m and 3m in length were left as is.

The Harrier Model (1512_HRM) has used a total of 651 drill holes with 79 RC (12%) and 572 diamond holes (88%).

Table 14-23 includes Descriptive model statistics for the Harrier Model (1512_HRM).

Table14-23Descriptive statistics for theHarrierModel(1512_HRM).

Model:	1512_HRM		Descriptive Statistics					
Date:	Dec-2015							
Variable	Data Type(s)	Numberof Samples	Minimum sample grade g/t	Maximum sample grade g/t	Mean grade g/t	StdDev g/t	Variance g/t2	Coeff of Var
Code 6 Splay LG								
AuRaw	DD	500	0.01	27.10	3.24	3.82	14.59	1.18
Au2.0mComposites	DD	310	0.01	24.60	3.14	3.47	12.04	1.11
CodeHarrier								
AuRaw	DD	958	0.01	45.30	6.84	5.93	35.16	0.87
Au2.0mComposites	DD	475	0.28	28.27	6.88	4.69	22.00	0.68
Code21HarrierBase								
AuRaw	DD	93	0.06	31.00	7.01	6.85	46.92	0.98
Au2.0mComposites	DD	43	0.59	26.54	7.07	5.53	30.58	0.78
Code22HarrierLink								
AuRaw	DD	29	1.36	10.70	4.08	2.46	6.05	0.60
Au2.0mComposites	DD	17	2.11	7.94	3.76	1.50	2.25	0.40
Code24HarrierHW								
AuRaw	DD	247	0.03	31.60	7.32	5.81	33.76	0.79
Au2.0mComposites	DD	120	0.04	20.11	7.28	4.50	20.25	0.62
Code25HarrierSplay								
AuRaw	DD	392	0.01	26.90	5.77	4.14	17.14	0.72
Au2.0mComposites	DD	189	0.46	20.80	5.60	2.94	8.64	0.53
Code30Osprey								
AuRaw	DD	818	0.01	29.80	6.37	5.08	25.81	0.80
Au2.0mComposites	DD	422	0.02	27.32	6.25	4.26	18.15	0.68
Code31OspreyBase								
AuRaw	DD	78	0.09	24.70	5.61	4.60	21.16	0.82
Au2.0mComposites	DD	37	0.60	14.53	5.53	3.31	10.96	0.60
Code 32 Osprey Link								
AuRaw	DD	129	0.07	16.80	4.87	3.29	10.82	0.68
Au2.0mComposites	DD	71	0.73	10.66	4.89	2.30	5.29	0.47
Code35OspreySplays								
AuRaw	DD	257	0.10	24.00	5.34	4.12	16.97	0.77
Au2.0mComposites	DD	142	0.20	15.75	5.17	3.11	9.67	0.60

Table 14-24 includes composite length statistics for the composite file used in the Harrier Model.

Table 14-24 Composite statistics by composite length for the (1512_HRM) Harrier Model

Composite length	Number	% of comps	mean length (m)	mean grade (g/t Au)
< 1.0m	183	10.0%	0.63	4.50
≥ 1.0m and < 2.0m	573	31.4%	1.34	5.27
≥ 2.0m	1071	58.6%	2.11	5.94
Total	**1827**	**100%**	**1.72**	**5.59**

No Au top cuts were imposed in the 1512_HRM Model. However, similar to the Phoenix and Central Models, search limiting was used to contain 12 - 20g/t Au data to interpolation distances less than the primary search distance for a given domain.

Variography

The Harrier Model shares many common elements with the Phoenix Model. During 2015, very little Mineral Resource was added to Harrier domains either by re-interpretation from mapping and sludging or by the very small amount of drilling added. Variography was conducted on the DOMAIN=20 Harrier domain in 2013 (ore wireframe) and variography was optimized using known geological interpretation. A sub-set of the domain data was used that favored longer ranges, i.e. where fault segments of controlling faults have oblique/parallel structural settings. The variogram and search parameters for the Harrier (1512_HRM) Model domains are summarized in Table 14-25 and Table 14-26, both with respect for gold but also for sulphur and NCC (non-carbonate carbon).

Table14-25 Variogramparametersused fortheHarrierModel goldestimation.

GOLDVARIOGRAMPARAMETERTABLE										
1512_HRM(HarrierModel)										
AREA	ZRotation	XRotation	YRotation	Nugget	1st Rotation Spherical Sill	Range (y)	Range (x)	Range (z)	ZRotation	XRotation
	meds rotation								*meds rotati*	
Code 6 Splay LG	5	5	55	7.00	1.00	10	10	5	5	5
Code 6 Splay LGvar 2	355	16	55	7.00	1.00	10	10	5	355	16
Code20Harrier	23	38	44	6.41	15.54	73	45	20	23	38
Code 20 Harrier var 2	77	51	10	20.93	10.10	65	20	10	77	51
Code20Harrieroutlierestimation	70	60	5	20.93	10.10	7	7	7	70	60
Code21HarrierBase	8	30	50	2.52	2.02	10	5	5	8	30
Code 21 Harrier Base var 2	70	50	5	2.52	2.02	10	5	5	70	50
Code22HarrierLink	10	1	60	2.52	2.02	10	5	5	10	1
Code24 HarrierHW	5	20	66	3.90	3.00	45	20	5	5	20
Code25HarrierSplay	150	20	-40	2.52	2.02	10	5	5	150	20
Code 25 Harrier Splay var 2	80	58	10	2.52	2.02	10	5	5	80	58
Code25HarrierSplayoutlier estimation	80	58	10	2.52	2.02	7	7	7	80	58
Code30Osprey	355	20	50	2.52	2.02	10	5	5	355	20
Code 30 Osprey var 2	114	47	-20	9.62	6.60	28	30	13	114	47
Code30Ospreyoutlierestimation	114	47	-20	9.62	6.60	7	7	7	114	47
Code31OspreyBase	18	30	50	2.52	2.02	10	5	5	18	30
Code 31 Osprey Base var 2	70	50	15	2.52	2.02	10	5	5	70	50
Code 32 Osprey Link	100	48	0	2.52	2.02	10	5	5	100	48
Code35OspreySplays	355	23	47	2.52	2.02	10	5	5	355	23
Code 35 Osprey Splays var 2	3	23	47	2.52	2.02	10	5	5	3	23

Table14-26 Variogram parameters used for the Harrier Model Sulphur and NC Cestimation

SULPHUR/NCC VARIOGRAM PARAMETER TABLE											
1512_HRM(Harrier Model)											
AREA	ZRotation	XRotation	YRotation	Nugget	1st Rotation Spherical Sill	Range (y)	Range (x)	Range (z)	ZRotation	XRotation	YRotatio
	meds rotation								*meds rotation*		
D99WasteSulphur	355	20	50	0.30	0.19	12	10	5	355	20	50
D98DomaallSulphur	355	20	50	0.30	0.19	12	10	5	355	20	50

D98DomaAllNCC	0	20	50	0.01	0.05	20	10	5	0	20	50
D99(Waste)NCC	0	20	50	0.01	0.05	20	10	5	0	20	50

14.2.4 Resource Modeling

Block Models

The Harrier Block Model was created to allow modeling of mineralization between 4700mN and 6250mN (Table 14-27).The XYZ block dimensions of 2m (east) by 10m (north) by 5m (RL) were used.

This block size was chosen after consideration of:

- Drilling with the intent to mine was conducted at a nominal density of 25m by 25m spacing, although some areas of the Harrier Mineral Resource are drilled to 12.5m spacing;
- Variogram model ranges between 10m to 30m;
- Typical mineralization width of 4m to 8m; and
- Likely underground mining methods (Smallest Mining Unit).

Table 14-27 Harrier Block Model extents and cell size.

Model Extents	Minimum	Maximum	Cell	Dimension (m)
Northing (m N)	4,700	6,250	X Direction (East)	2
Easting (m E)	1,400	2,100	Y Direction (North)	10
Reduced Level (m RL)	4,200	5,200	Z Direction (Vertical)	5

The Harrier Block Model used Ordinary Kriging to interpolate grades without a composite top cut.

Search Criteria

Search Criteria methods and justification within the Harrier Block Model are the same as those used for the Central Area.

Similarly to the 1512_PRM (Phoenix Model) the Harrier underground model employed:

- No direction de-clustering.
- A maximum of one to six composites per drill hole, dependent on intercept width and drill hole spacing. The restriction was enforced to minimize single-hole block estimation.
- A lower composite outlier 'soft boundary' , used to limit the effect of grade variability influencing higher grade and more densely supported area.

Search ellipses, shown in Figure 14-32, depict the maximum range extents that composites can be used to estimate a block. Search parameters for the Harrier Block Model are provided in Table 14-28. Model test work combined with visual validation of mapping, culminated in employing e a shallow plunging search informing distal blocks with as few as two composites within the DOMAIN=20 Harrier domain (ore wireframe). This was combined with a steep plunging overprint that required three composites. The net effect allowed poorly informed blocks on the periphery of the DOMAIN=20 Harrier domain to be estimated whilst areas that were well informed had to fulfil the stricter criteria to influence block grades. In addition, Table 14-28 includes the search parameters for the lower composite outlier soft boundary which was employed by using a sub domain inside the DOMAIN=30 Osprey was run at the end of the interpolation process as a local overprint of the DOMAIN=30 domain confined to the sub domained zone.

Figure 14-33 shows the 1512_HRM Block Model with respect block size, Resource Domains and the December 2015 Mineral Reserve.
Similarly to the Phoenix Block Model, the steep search ellipse criteria, within the Harrier Block Model, limits the strike influence of the high-grade shoots. The steeper plunging estimation direction is in line with geological plunge intersection trends between the parallel/ oblique mineralized structures and intersecting splays. This shoot geometry is usually associated with wider zones of mineralization increasing the number of composites available at a higher grade than is seen towards the edges of the mineralization.

Table14-28 Search parameters for the Harrier Model.

SEARCH PARAMETER TABLE

1512_HRM(HarrierModel)

DOMAIN	y axis	x axis	z axis	Rotation (z)	Rotation (x)	Rotation (y)	min. samples 1st search	max. samples 1st search	y axis	x axis	z axis	Rotati (z)
	1st/3rd search distance			rotation			sample number definition		2nd/4thSearchDistance			
Code 6 Splay LG	125	125	60	5	5	55	1	35	60	30	20	355
Code20Harrier	90	75	40	23	38	44	2	35	60	30	15	77
Code20Harrieroutlierestimation	7	7	7	70	60	5	1	35				
Code21HarrierBase	85	85	50	8	30	50	1	35	60	30	20	8
Code22HarrierLink	35	25	10	10	1	60	1	35	45	25	5	1
Code24 HarrierHW	90	65	30	5	20	66	90	35	50	40	20	5
Code25HarrierSplay	90	50	40	150	20	-40	1	35	80	30	20	80
Code 25 Harrier Splay outlier estimation	7	7	7	80	58	10	1	35				
Code30Osprey	100	60	40	355	20	50	2	35	90	40	30	114
Code30Ospreyoutlierestimation	7	7	7	114	47	-20	1	35				
Code31OspreyBase	90	60	25	18	30	50	2	35	80	40	30	70
Code 32 Osprey Link	70	55	35	100	48	0	2	35				
Code35OspreySplays	90	70	30	355	23	47	1	35	45	45	20	3

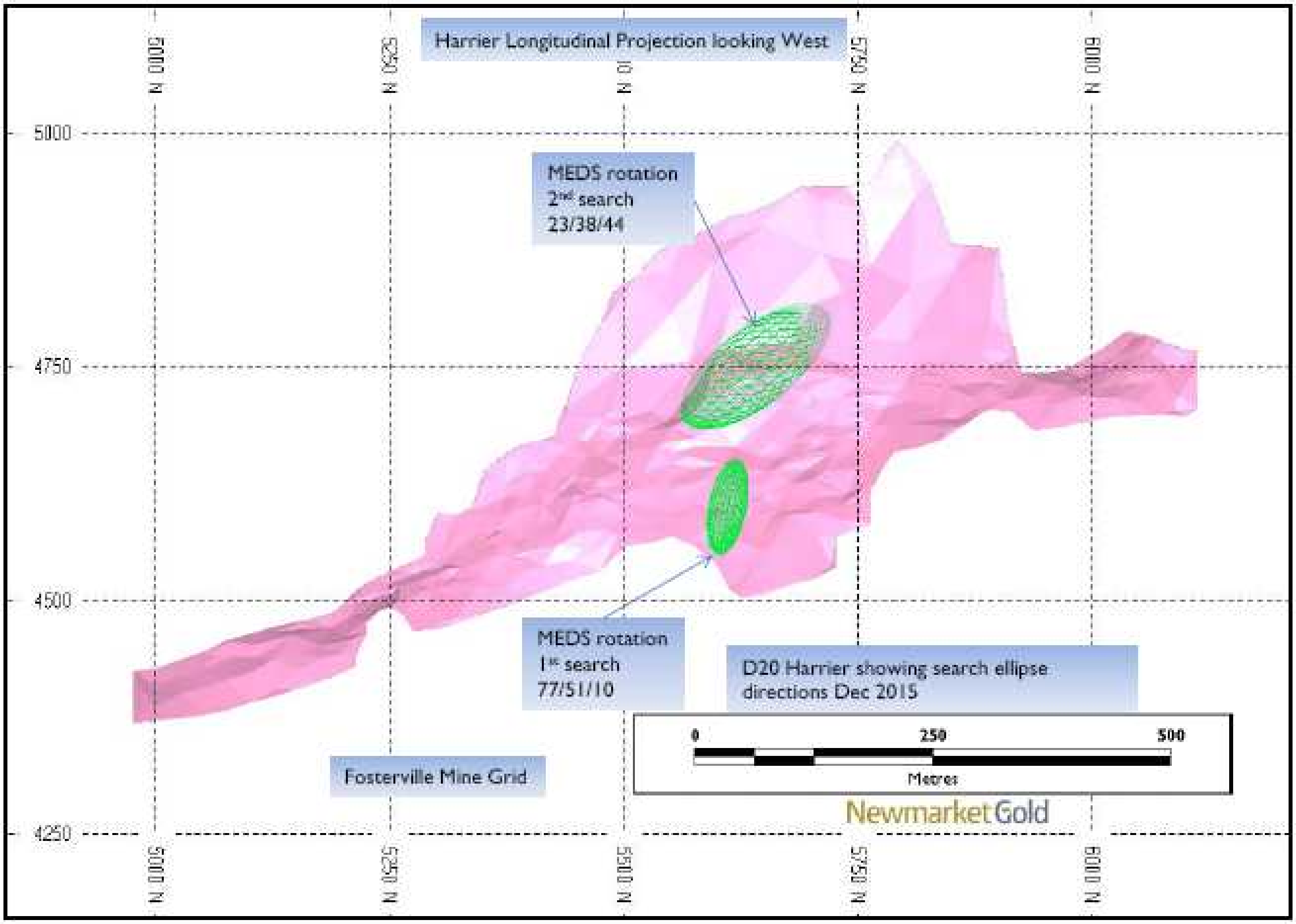

Figure 14-32 Search Ellipse for DOMAIN=20 Harrier.

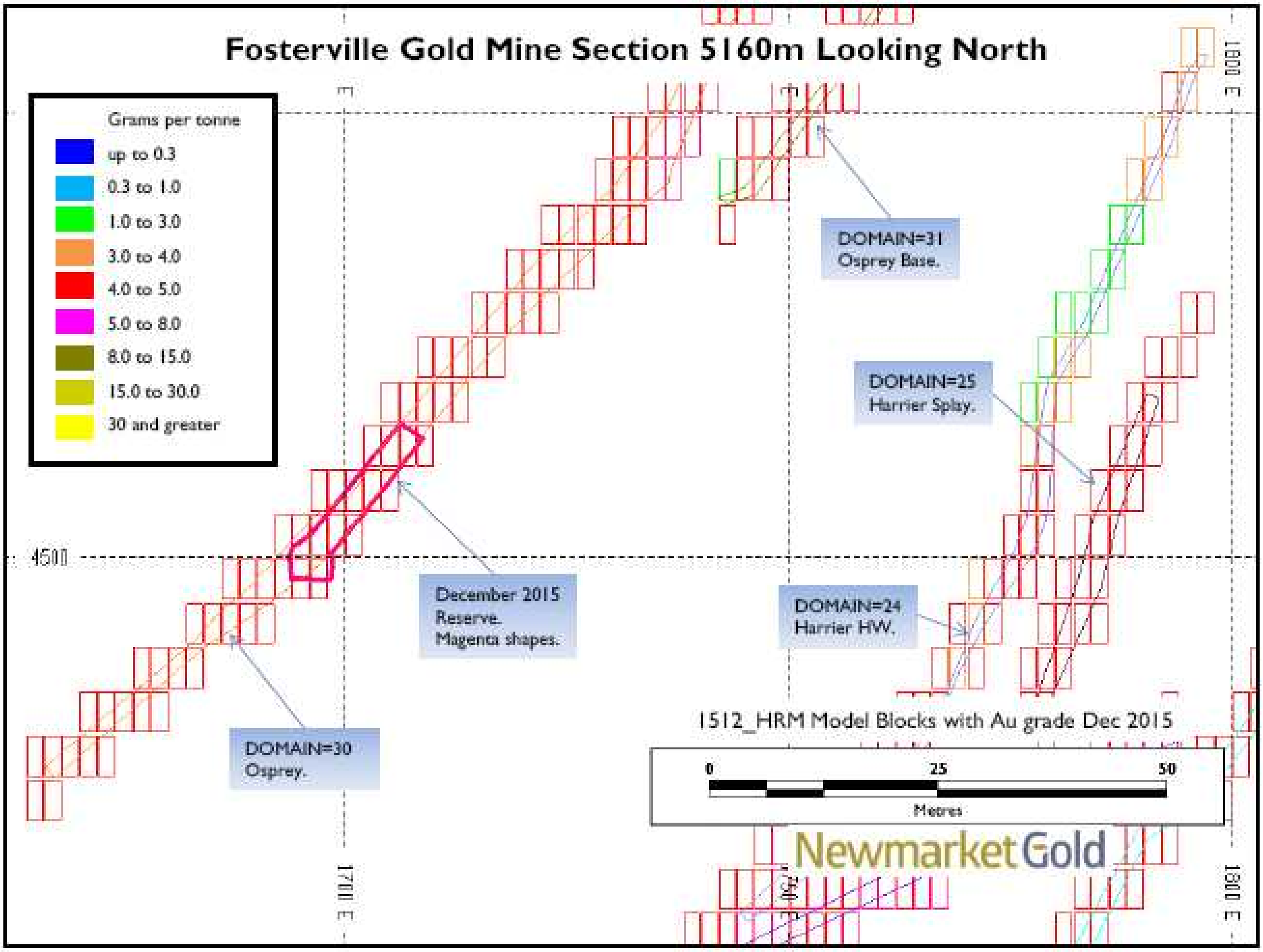

Figure 14-33 Section through the 1512_HRM Harrier Model (5160mN), showing the Osprey Mineralization Domain.

Bulk Density

Bulk density data obtained from exploration diamond core testing within the model area showed no material difference from density data obtained in the Central Area Models. Consequently, bulk density values were assigned to the Harrier Block Model according to material type using values from data collected in the Central Area (Figure 14-27). As mining continues below the 4500mRL, further density data will be collected to compliment density measurement taken from similar levels within the Phoenix area.

14.2.5 Mineral Resource Classification

The mineral resource classification for the Harrier Block Model is the same technique applied as within the Central Area. Figure 14-34 illustrates the Harrier Model Resource classification.

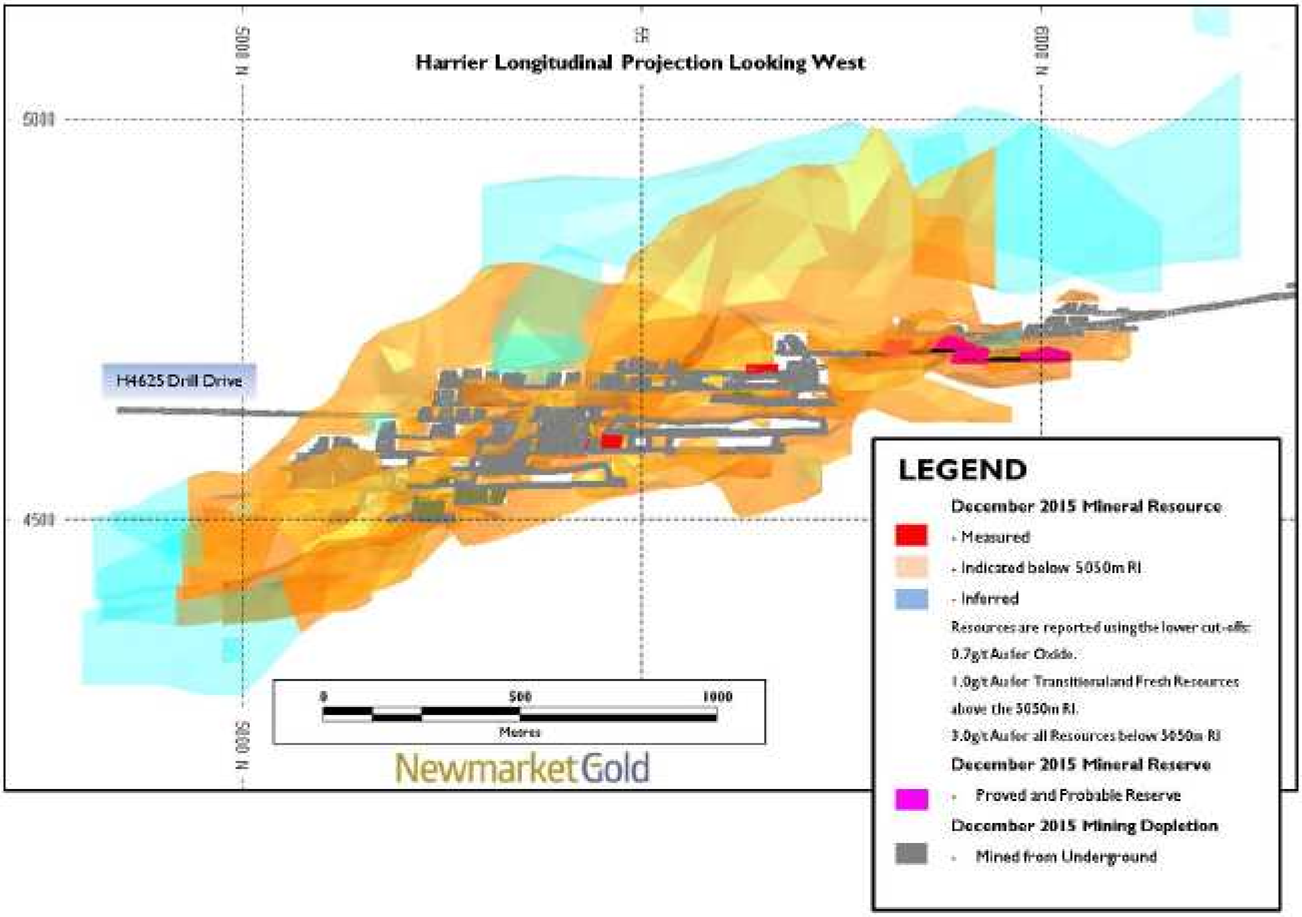

Figure 14-34 Longitudinal projection looking west showing resource classification for the Harrier Model.

14.2.6 Results

Results for the Mineral Resources contained in the Harrier Area are provided in Table 14-1.

14.3 Fosterville-Hunts Area

The Fosterville-Hunts Model is located to the north of the Central Area and is defined as the zone between 10,000mN and 11,500mN (Figure 14-1) and conveniently extends over Fosterville and Hunt's oxide pits.

14.3.1 Area Discussion and Results

The controlling structural features from west to east include: the moderately west dipping Hunt's Fault, several footwall splays and the Fosterville Fault (Figure 14-35). The geology of the area was assessed by Fosterville staff, later reviewed by Stephen King (King, 2007) and mineral resource modeling undertaken by Kerrin Allwood (2008).

The gold mineralization in the Fosterville-Hunt's area was historically mined for oxide gold and in the 1990's mining for oxide heap leach material created the Fosterville and Hunt's oxide pits.

However, since 2010 flotation in-pit tailings has and is being placed into the Fosterville and Hunt's pits. This tailings placement has resulted in no Mineral Resources being reported from the Fosterville-Hunts area for 2015.

It is the opinion of the Authors that the placement of tailings within the Fosterville and Hunts pits currently impedes reasonable prospect for economic extraction of the mineral occurrence which lies directly below these pits.

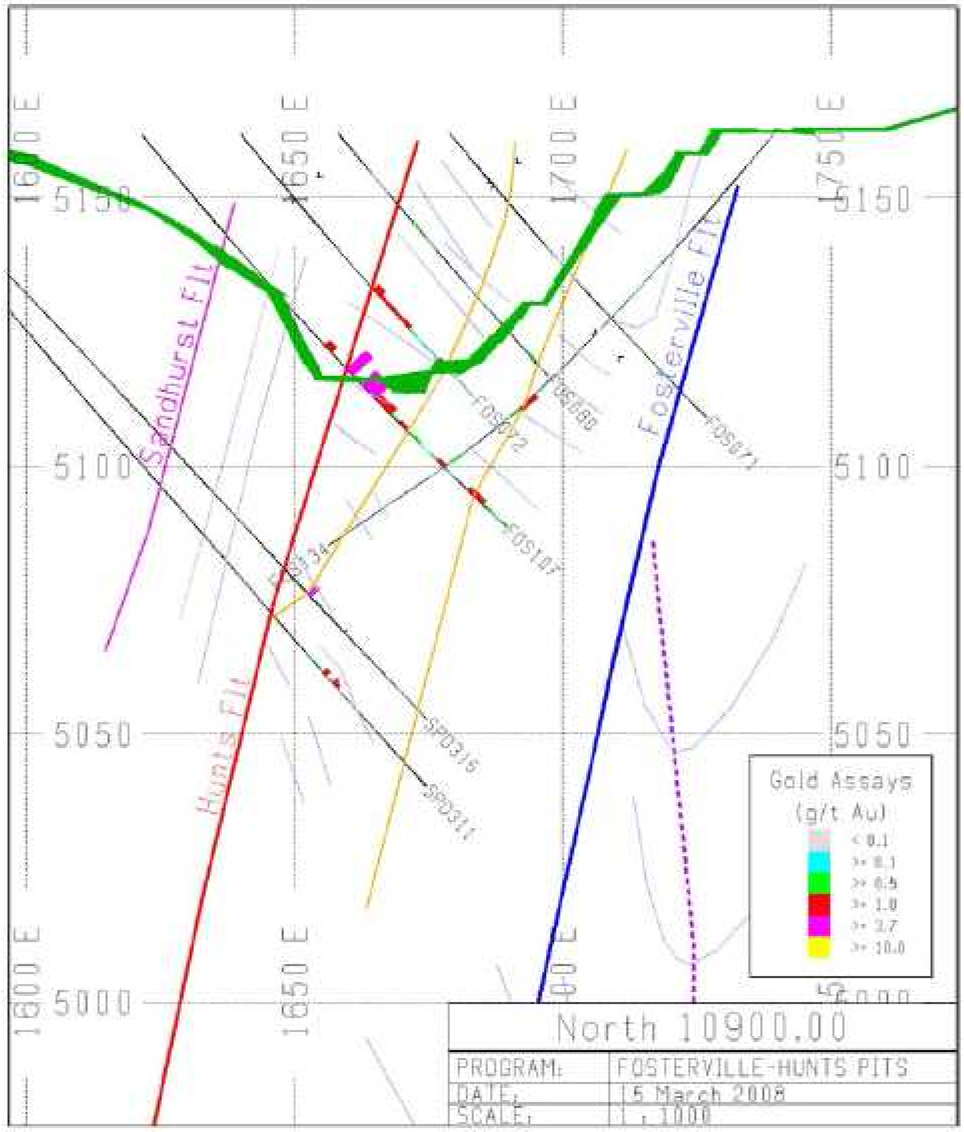

Figure 14-35 Geological Cross section 10,900mN through Hunt's Pit.

Shown are the relationships between the Hunt' s Fault, bedding and the set of splays that strike obliquely to the fault.

14.4 Daley' s Hill Area

The Southern Model spans from the Harrier Pit area to Daley' s Hill Pit, close to the southern margin of the Fosterville Mine Lease (MIN5404) as shown in Figure 14-1.

The Southern Model was in existence before the initial Harrier Mine Model became operational. Where there is overlap between the Harrier Model and Southern Model, the Harrier Model is used in preference for Mineral Resource reporting with the only exception being the Daley' s Hill Pit area (south of 5300mN and above 4800mRL), where Southern Model has been used. Only the Daley' s Hill area is discussed in detail in the following sections.

14.4.1 Area Geology

Within the Southern Model area, the controlling features include the Fosterville Fault and the footwall Harrier suite of faults, which have variable reverse offsets and a total reverse displacement of about 200m.

Reverse movement on the Fosterville Fault lessens from north (100m+) to south (~10m at Daley' s Hill) and becomes less important southwards with respect to mineralization. At Daley's Hill the Fosterville Fault is un-mineralized and passes to the west of the oxide pit.

The east-west folding in the area varies from gently southerly plunging in the north to moderate southerly plunging at Daley' s Hill in the south. Fold plunge is important as the mineralized west dipping fault geometry is controlled by eastern limbs of syncline fold plunges where the faults become un-mineralized "bedded" LQ features.

At Daley' s Hill, the Daley' s Hill Fault has an associated 10m of reverse fault movement and localizes the bulk of gold mineralization. Lesser well mineralized east-west structures occur in the eastern parts of the pit and several other poorly defined hangingwall mineralized fault structures are present in the western portions of the pit.

Daley' s Hill is unusual in that late stage free primary gold, in association with stibnite-quartz, is noted in two diamond holes (DALD05 and DALD06). The mineralized structure ("Wagon Wheel") is restricted to an 80m strike extent, but is untested at depth.

The geology of the Southern Model area was reviewed by independent consultant Stephen King in 2004 (King, 2004) and the northern parts again in 2006 (King, 2006). Rod Boucher (geological consultant, Linex Pty Ltd Geological Consultation) has also contributed much to the stratigraphic-structural understanding of the area. Geological interpretation is also reported by Reed (2007).

14.4.2 Geological Models

Geological modeling undertaken is essentially identical to that used for the Fosterville-Hunt' s and Robbin's Hill Models. Several iterations of mineral resource modeling of the Southern Model were undertaken and reported in Hitchman (2006). A review of the 2006 resource work was undertaken by Scott Jackson from QG Consultants (Jackson, 2007).

14.4.3 Domains

Domaining of the Daley's Hill area was based on geological structure, orientation, material types and variography. The structures and material types include:

- Daley' s Hill N-S Faults
- Daley' s Hill E-W Faults
- Materials (Oxide, Transitional and Fresh)

Mineralization domains were created by firstly using a nominal 0.2 to 0.5 g/t Au outer limit for sectional strings in weathered areas and 0.5 to 1.0 g/t Au in unweathered mineralization. These values used reflect natural breaks to the mineralization.

The strings were then linked or extruded to form a three dimensional wireframe domain. The strings were generally extruded a maximum of half the drill spacing. This varied from as little as 5m, in well drilled pit locations, to 50m, where mineralization extended over several 100m spaced drill sections.

Daley' s Hill Domains

The Daley's Hill area (Figure 14-36toFigure 14-38) has three separate northerly trending sub-vertical to westerly dipping mineralized domains:

DH Main Fit
DH Wagon Wheel
DH West Area

The domains have variable strike lengths (between 50-650m), dips (-50°W to -90°) and exhibit ~20° southerly plunges.

A domain (DH Syncline) has also been generated that encompasses mineralization associated with the Daley' s Hill Syncline. The syncline axial plane trends grid NNE with a 45° plunge towards the south and is located in the far northern position of the existing Daley' s Hill Pit.

East-west mineralized structures occurring in the eastern parts of the pit are footwall to the main Daley' s Hill N-S structure. The Daley' s Hill E-W domain (DH Campaspe) comprises four separate structures, which trend 060° and dip 80°N.

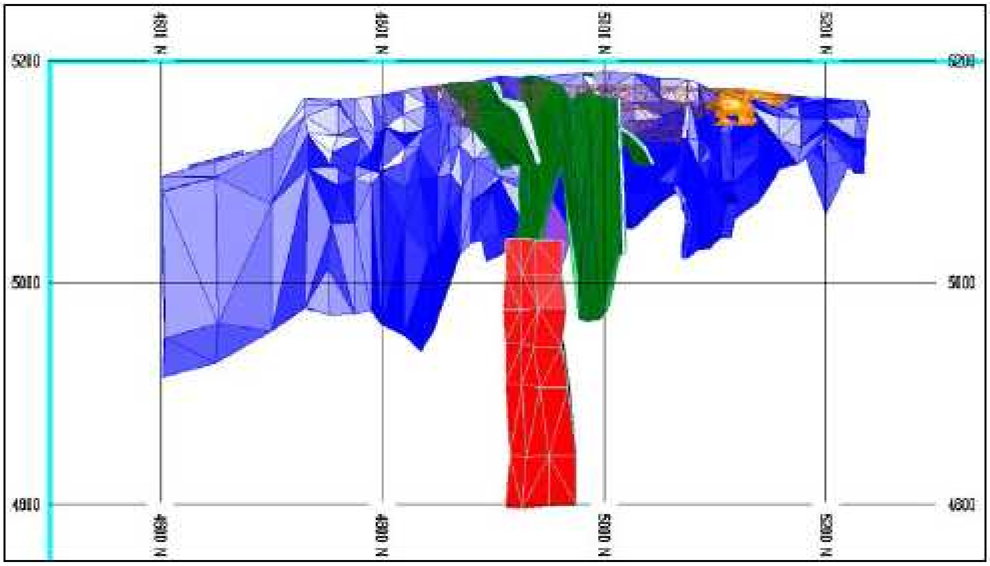

Figure 14-36 Longitudinal projection looking west at Daley' s Hill area.
Shown are model boundary (light blue), mineralized solids (red, blue and green) and Daley' s Hill oxide open pit (brown wireframe)

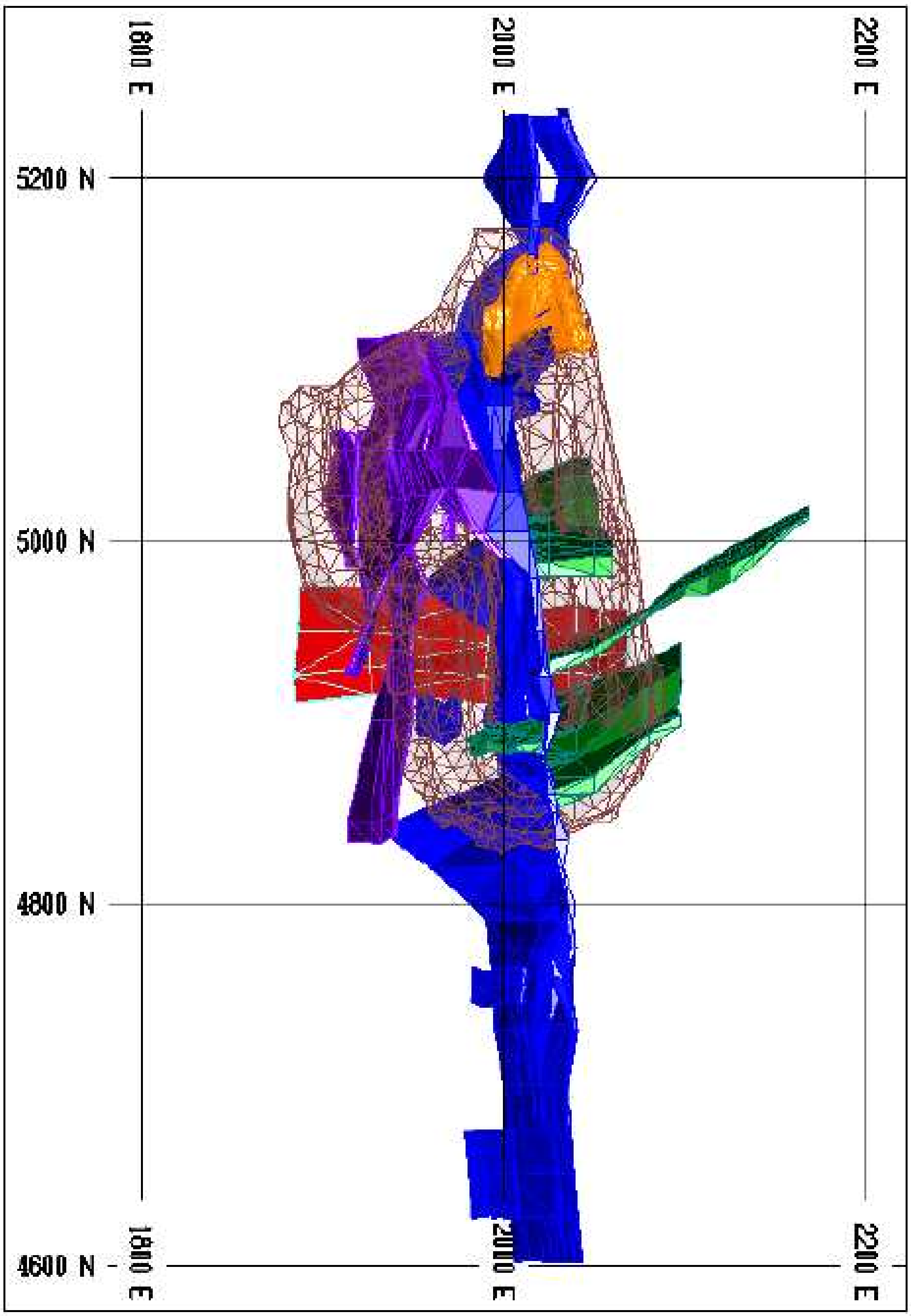

Figure 14-37 Plan view of Daley' s Hill mineralization.
Shown are the DH Main Flt (blue), DH Wagon Wheel (red), DH West Area (purple), DH Syncline (orange), DH Campaspe (green) and oxide open pit (brown wireframe)

Figure 14-38 Oblique northerly view of Daley' s Hill mineralization.
Shown are DH Main Flt (blue), DH Wagon Wheel (red), DH West Area (purple), DH Syncline (orange) and DH Campaspe (green) Mineralization and oxide open pit (brown) wireframes

Three 'material' domains were constructed, similar to that described previously for the Central Area Model. The domains are:

- Oxide (sulphide minerals completely oxidized, Fe-carbonates largely oxidized)
- Transition (sulphide minerals may be partially oxidized, includes zones of mixed fresh and oxide), and
- Fresh (sulphide minerals completely un-oxidized)

The Transitional domain lower boundary is only an approximation because there is insufficient logging of the base of transition to allow a reasonable interpretation of this surface over the entire Southern Model. The base of transition was taken to be 5110mRL after comparison with drill data and results from open pit mining in the area.

Separate material domains were constructed for transitional and fresh materials and coded into the Southern Model for inventory and metallurgical recovery study purposes. However, during block model interpolations, drill assays coded as transitional and fresh material types are treated as if they are the same material type.

14.4.4 Drilling Data

The drilling quality is variable in the Southern area and includes:

- RAB – Rotary air blast;
- Reverse circulation – Cross over hammer and face sampling hammer variants; and
- Diamond core – HQ and NQ2, often with RC pre-collars.

During drillhole data extraction for resource interpolations, the omission of RAB holes and one diamond hole was required owing to low quality sample techniques and incomplete assaying respectively. MineSight drill views were the primary tool used to identify data problems.

Subsequent to the drill data review process assay data were:

- Imported from the acQuire Exploration databases into MineSight using customizable parameter screens; and
- Coded for mineralization using 3D gold wireframe solids.

Within the oxide open pit areas, the historical 5m blast holes are vertical and generally had one sample collected over a 5m length. These holes were used to aid interpretation, but were not used during subsequent Kriging owing to sample quality and that the 5m sample lengths were in excess of the desired 2m composite lengths.

Compositing and Coding

Compositing and coding of drill holes was undertaken similar to the Central Area.

Variography

In the Daley' s Hill area where drill spacing is nominally on 10-20m is available, variography work demonstrates relative nugget effect values of 50% and most of the variance in the first ~30m. The variogram models closely follow the expected geological controls with 20° southerly plunging shoots in 70° west dipping faults.

14.4.5 Mineral Resource Modeling

Block Models

The Southern Block Model (Southern Model) was originally created to allow modeling of gold mineralization south of 7,400mN to the southern end of the Fosterville Mine Lease. However, as mining advanced southwards, the use of the Southern Model has diminished, such that it is only being used for reporting Mineral Resources in the Daley' s Hill area.

The Southern Model XYZ block dimensions of 4m (east) by 10m (north) by 5m (RL) were used. This block size was chosen after consideration of the maximum drilling density (25m by 15m), mineralization geometry (typical mineralization width of 3m to 8m) and probable open pit mining methods.

Search Criteria

Gold grades were interpolated into blocks meeting the following block criteria:

- Greater than 1% of the block volume is inside one of the domain envelopes;
- blocks whose search ellipse includes at least five composites; and
- Blocks whose material code is set to Fresh (1), Transitional (2) or Oxide (3).

Similarly, only composites meeting the following criteria are used to interpolate any one block:

- All composites (to a maximum of 30 composites) within the search ellipse dimensions and search area limits outlined in the table below;
- Where more than 30 composites lie within the search ellipse the 30 closest composites in ellipsoid space are used;
- Maximum of six composites are used from any split quadrant of the search ellipse (a split-quadrant is 1/8th of the search ellipse dividend in the major, intermediate and minor ellipse axes); and
- The CODE1 and MATL values of both the composite and the block must match (i.e. only fresh composites from are used to interpolate a fresh block and vice versa for oxide).

The search ellipse orientations follow the kriging axes. The search ellipse dimensions allow the block being interpolated to 'see' two sections along strike and two holes up or down dip.

Bulk Density

The bulk density profile (Figure 14-27) established for the Central Area was taken as being appropriate for the Southern Model given the similar rock types, levels of oxidation and identical mineralization and gangue mineralogy. Deep drilling in the Central Area and Harrier Area has supported the inclusion of a bulk density value of 2.78 t/m^3 for material below 4500m RL. However, as the mineralization at Daley's Hill is shallower than 4500mRL, reporting of Resources for this area from the Southern Model is unchanged.

14.4.6 Mineral Resource Classification

Three solids were created enclosing regions of geological confidence (Measured=1, Indicated=2 and Inferred=3) and these three regions were used to code the Mineral Resource category item in the block model. The solids generally enclose areas of approximately equally spaced drilling, but also allow areas where there is reduced confidence in the geological interpretation to be reported to a lower confidence category.

In areas of the Southern Model at depth below and to the north of the Daley's Hill Pit, the diamond drilling is on nominal 100m north spaced drill sections with 50m down dip holes spacing, and for this drill density the mineralization is broadly classified as Inferred Mineral Resource. Beneath the open pits where the drill spacing is reduced to 10m – 20m north by 10m – 15m east, mineralization is classified as Measured Mineral Resource with a halo of Indicated Mineral Resource.
The Daley's Hill east-west structures are not well understood and as such this mineralization is classified as Inferred Mineral Resource.

14.4.7 Results

Results for the Mineral Resources contained in the Southern Model are provided in Table 14-1.
Small oxide gold resources exist in the Daley' s Hill area and are confined along strike from the previously mined open pit in the top 40m. The bulk of the sulphide mineral resources reported from the Daley' s Hill area within the Southern Model are based on 100m by 50m spaced diamond drilling supplemented by closer spaced, but lower quality face and cross over RC drilling. Infill drilling will be required to increase resource confidence from an Inferred Mineral Resource category.

14.5 Robbin' s Hill Area

Robbin's Hill Area lies northeast of the Central Area and contains the O'Dwyer's, Robbin's Hill, Farley's, Sharkey's, Woolshed and Read's oxide pits as shown in Figure 14-1. The area can be defined as the zone east of 2,700mE, between 10,500mN and 14,000mN.
The controlling structural features in the area include a variety of north-trending west-dipping faults and failed anticline axes intruded by dykes. The Sharkey' s area is one of the few areas known on the lease where east dipping structures are significantly mineralized.
The geology of the area was assessed by Fosterville staff during diamond drilling activities between 2004 and 2007, reported by Reed (2007a) and reviewed twice by Stephen King (2005 and 2007). The area was also the subject of a study conducted by Chris Davis (Davis, 2006). Robbin's Hill Model resource modeling conducted by Kerrin Allwood and Simon Hitchman is reported in Allwood (2006) and Hitchman (2007). A further review of modeling in the Farley' s-Sharkey' s area is also reported in Allwood (2007). Following on from an open pit optimization study in March 2011 (Dincer, 2011) 5,257m of combined RC and diamond drilling was undertaken in the Robbin' s Hill Project area to test beneath and along strike from existing open pits. This drilling was for both open pit and underground targets occurred in the Robbin's Hill area until August 2012, during which resource modeling was undertaken.
A short-lived sulphide open pit mining operation was completed at the O'Dwyer's South pit in 2012 and is now the site for flotation tailing storage.
The Robbin's Hill Resource Model, with smaller contributions from a Farley's Resource Model and O'Dwyer's South Grade Control Model, form the basis of the Report on the Mineral Resources and Mineral Reserves of the Fosterville Gold Mine for the Robbin's Hill area.

14.5.1 Area Geology

O' Dwyer's to Robbin' s Hill Area – Geological Overview

The fault architecture of the O' Dwyer' s-Robbin' s Hill (ODW-RH) area is more complicated than that observed in the Fosterville Fault Zone which generally has structural complexity on only one side of the Fosterville Fault.

Three significant fold closures occur in the west of the area – the Robbin' s Hill Anticline and Syncline and the Trench Syncline (named after the trench it was exposed in). The latter is discussed further in the Farley' s-Sharkey' s section. The Robbin' s Hill Anticline and Syncline pair loses amplitude and wavelength southwards from a wavelength of around 100m in the north to become a small parasitic fold pair in the south of the Robbin' s Hill pit. The folds are complicated by fault structures and the anticline is intruded by a porphyry dyke (RH Porphyry) within the pit. The dyke intrudes along the axial plane and pinches out to the south. This is interpreted to coincide with the position where the fold pair has diminished and a consistently east dipping stratigraphy is present – the sub-vertical dyke is oblique to this stratigraphy and so cannot easily continue to the south.

There is considerable variation in fold plunge directions and amounts in the ODW-RH area. Mapped north plunging bedding-cleavage intersections are present in the north and west of the area whilst south plunging intersections occur in the rest of the area but the degree of south plunge can change across faults (the ODW South Pit also has north plunging intersections which complicates the geometry further). The changes in fold plunge have a strong bearing on the complexity of the fault system as structures are commonly parallel to bedding. There is a change from grid north trending structures to more NNE trending structures moving northward which is probably intimately associated with fold morphology.

Two NNE trending structures (now beneath backfill) have been interpreted from Robbin' s Hill Pit mapping at 12,800mN. A zone of faulting links between the two zones and controls ore blocks, which cross-cut bedding and may be interpreted as an extensional link structure in a component of sinistral shearing on the NNE faults.

The central and northern portions of the Robbin' s Hill Pit have not yet been modeled.

In the ODW South and North Pits the same west dipping fault structure is mineralized and has a curvilinear grid north trend. East of, and paralleling this fault, is an anticline structure which has mineralized porphyry dyke (ODW Porphyry) occupying the sub-vertical axial plane. The ODW porphyry occurs in the eastern portion of the ODW South pit and in the middle of the ODW Central Pit. Several west dipping mineralized faults occur on both sides of the ODW Porphyry and outcrop in ODW Central and Eastern Pits. The west dipping faults appear to stop at the ODW Porphyry (Figure 14-39).

Northeast trending unconsolidated Murray Basin clays, sands and gravels mask the Ordovician basement in the northwest and southeast parts of the Robbin's Hill Model area (see Figure 14-1)

Shown in Figure 14-39 are open pits (white), March 2011 AUD$1350 open pit optimization shells (pink) and mineralization domains (faults in blue colors and felsic dykes in green)

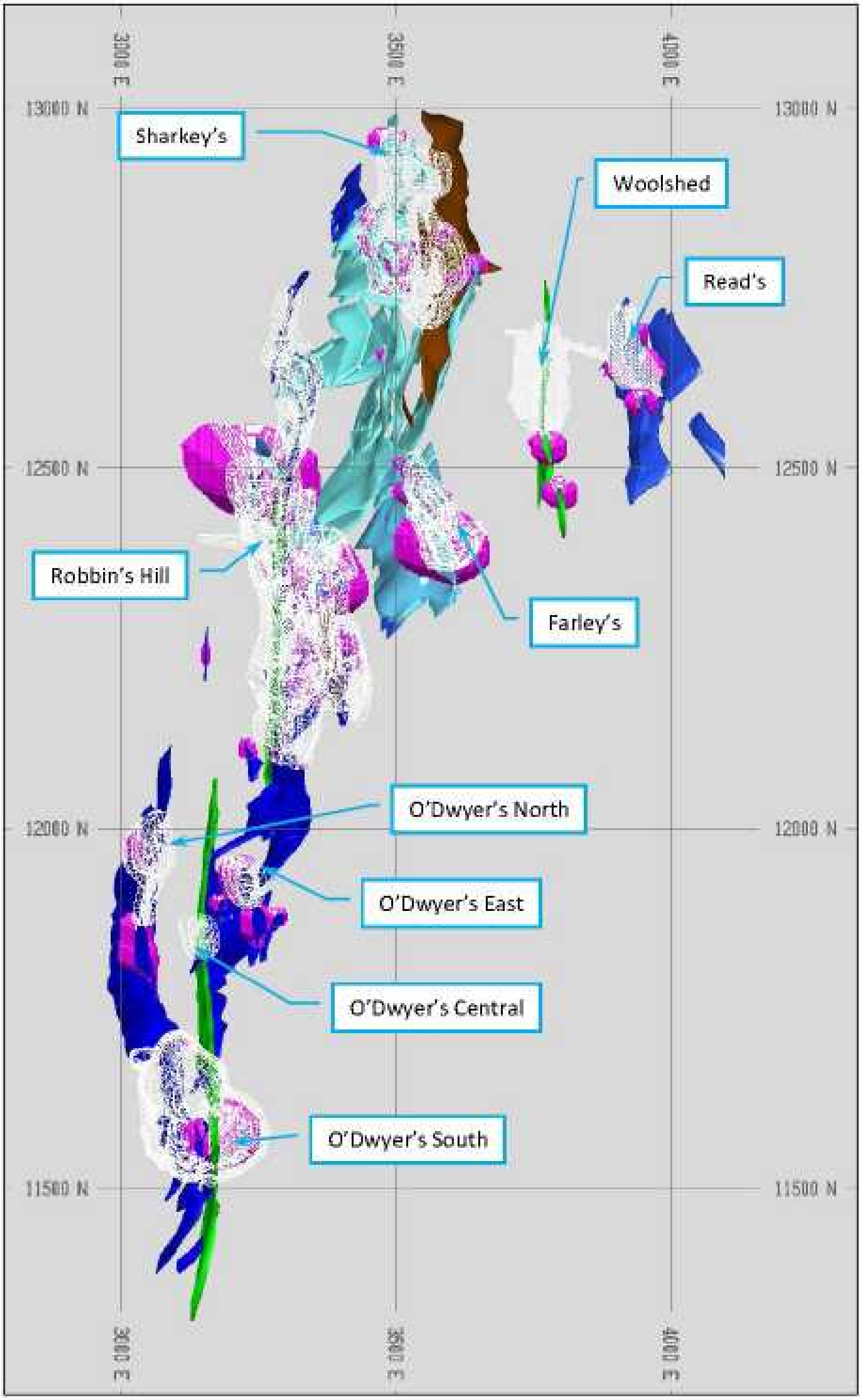

Figure 14-39 Oblique 3D perspective northerly view of ODW-RH Area.

Farley' s-Sharkey' s Area - Geological Overview

The Farley's Zone of mineralization is hosted by an anticline-syncline pair. These folds represent a set of parasitic east verging structures on the eastern limb of the Trench Syncline (Figure 14-40 to Figure 14-42). The Trench Syncline is a major north-south (350°) trending asymmetric syncline that plunges towards the south at about 15°. The eastern limb of the Trench Syncline has a shallower dip than the western limb and as a consequence the parasitic folding and bedding on the eastern limb has an observed strike of approximately 340-350°. This asymmetry of the folding accounts for the atypical strike of the Farley' s Pit in plain view.

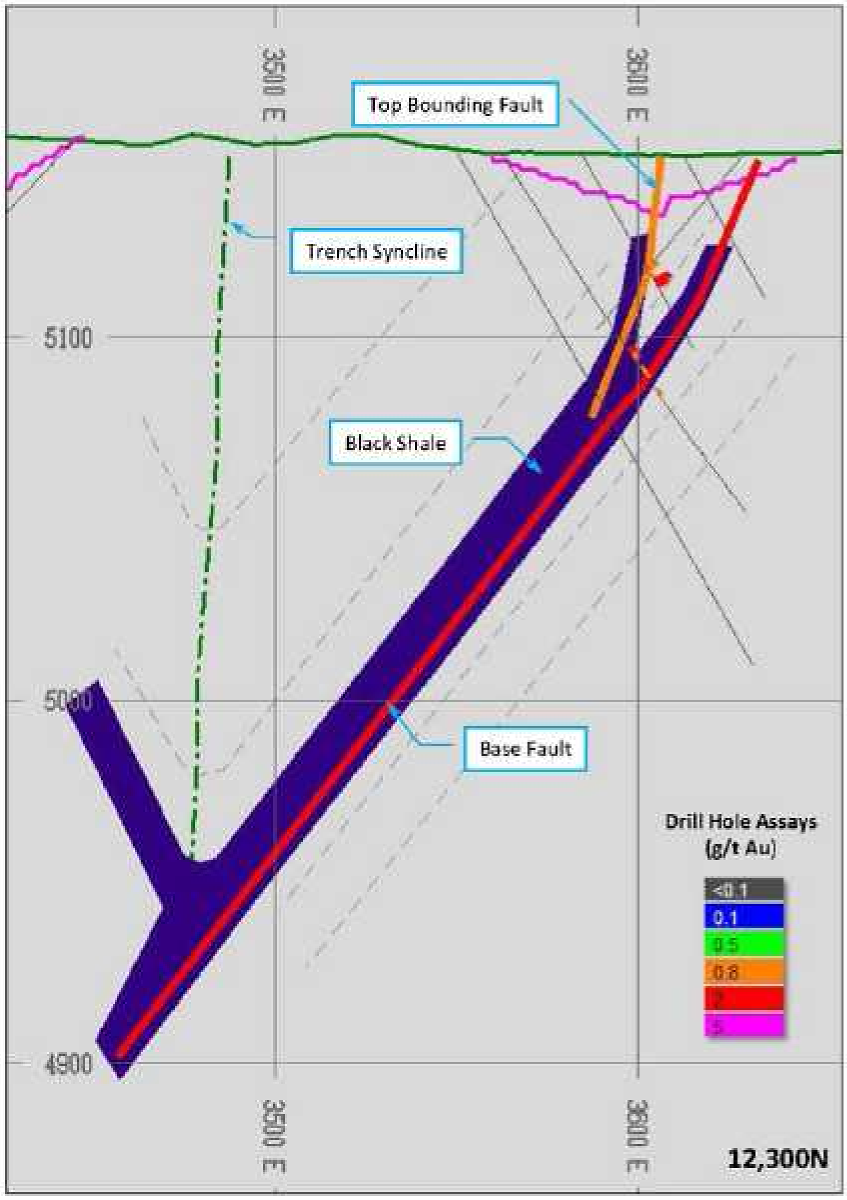

Figure 14-40 Cross-section for 12300mN, looking grid north.

On 12,300mN, the Base Fault (red) is bedding parallel in the sedimentary black shale unit (blue) on the eastern limb of the Trench Syncline.

The Farley' s anticline and Syncline parasitic folds are developed beneath a sedimentary black shale unit that wraps through the Trench syncline. This black shale unit can be seen in the Robbin' s Hill Pit where it dips to the east. The stratigraphy above the black shale can be correlated between holes and appears to correlate with the stratigraphy above the black shale in the Robbin' s Hill Pit.

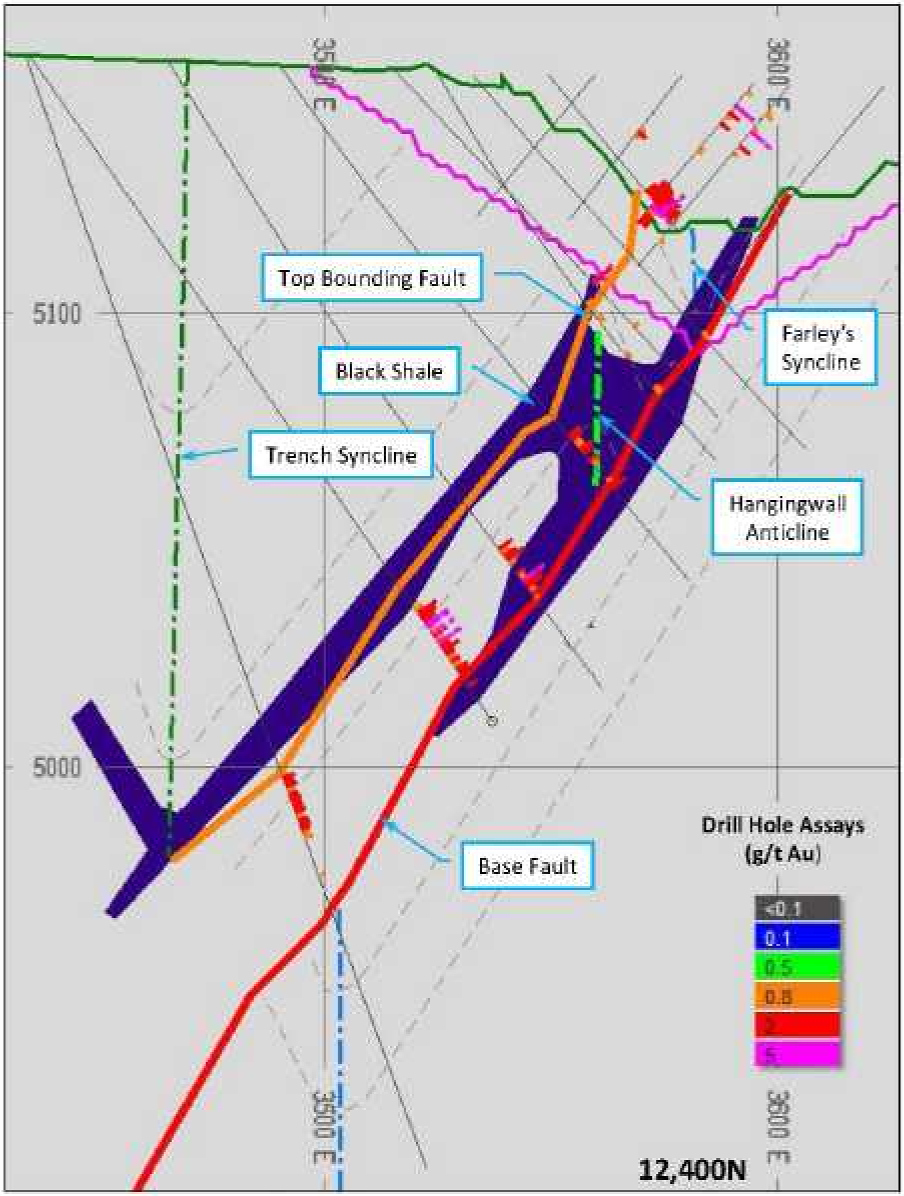

Figure 14-41 Cross-section for 12400mN, looking grid north.

On 12,400mN, parasitic folding (Anticline Hangingwall Splay and the Farley' s Syncline) is developed on the eastern limb of the Trench Syncline and is bounded by the Top Bounding Fault (orange) at the base of the black shale unit (Blue) and the Base Fault (red). Gold mineralization is confined to the area beneath the black shale (and hydrocarbon alteration) where it wraps through the anticline and syncline hinges. A conceptual open pit (completed in March 2011) outline is shown in pink.

The Top Bounding Fault is developed towards the base of black shale unit. This fault appears to have accommodated the movement in the black shale during folding and breaks out of the shale where it folds through the parasitic anticline and the Trench Syncline.

On the eastern side, the parasitic folds are truncated by the Base Fault. This is major north-south trending fault zone that is broadly conformable to the footwall but is discordant to the hangingwall, i.e. it separates the asymmetry of the western limb of the Trench Syncline from north-south trending west dipping bedding in the footwall. Given the form of this fault, it is interpreted as a major dislocation surface that has developed beneath the parasitic folding. The parasitic folding is therefore confined to the space between the Base Fault and the Top Bounding Fault in the black shale unit.

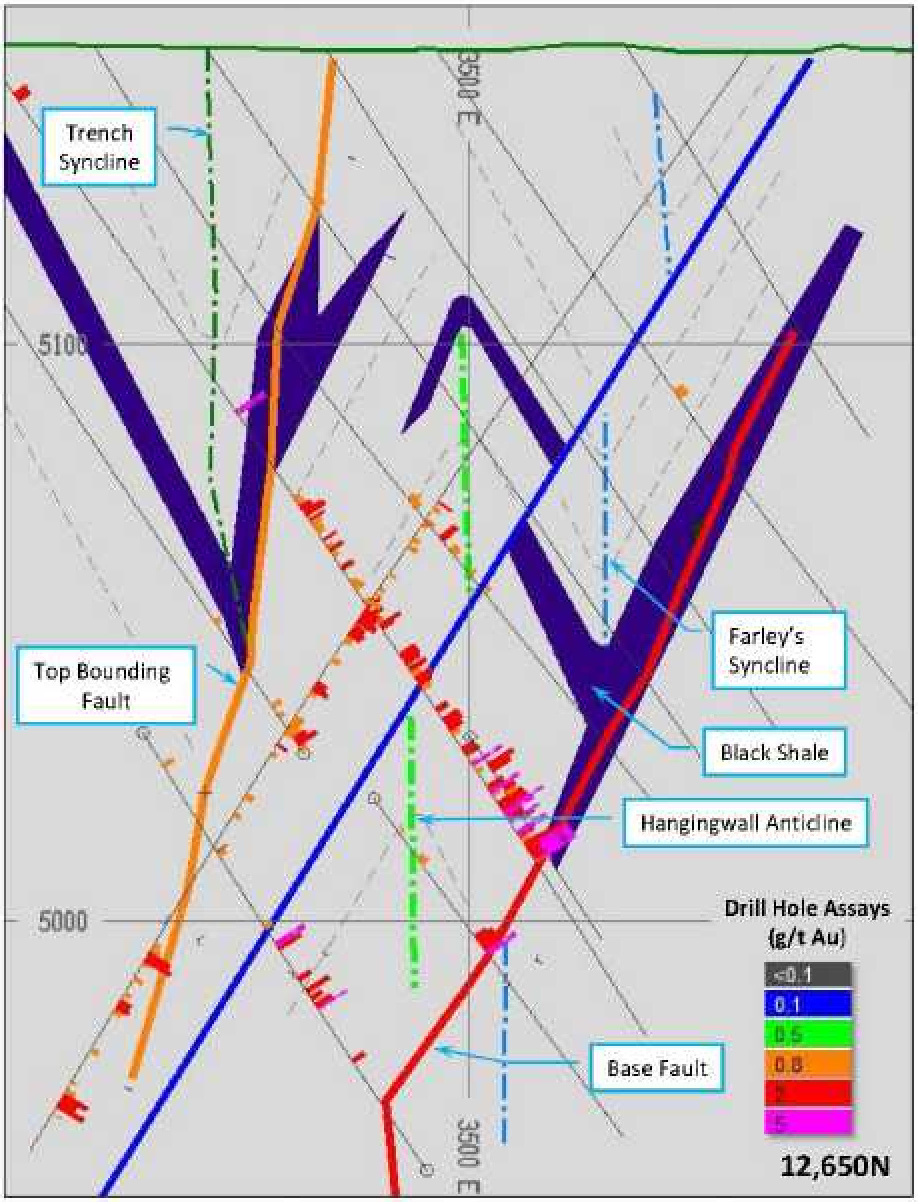

Figure 14-42 Cross-section for 12650mN, looking grid north.

On 12,650mN, the main sulphide zone associated with the faulted Farley' s Anticline steps up onto the faulted Trench Syncline. This corresponds with the Trench Syncline hinge being faulted out on the Top Bounding Fault. The parasitic folding observed to the south has now become more open and the Base Fault has steepened markedly with the Farley' s Syncline hinge, now seen in both the Footwall and Hangingwall of the fault. The kink zone in the Base Fault (and therefore the position of the Farley' s Footwall Syncline) controls the position of the high-grade zone intersected on the Base Fault.

The zone between the Base Fault and Top Bounding Faults represents a zone of increased deformation with shearing and quartz-carbonate stockwork veining developed between the two faults but focused on the two parasitic fold hinges. Within this zone, deformation has occurred mainly as puggy faulting with little actual displacement on any one fault. In the sandstone units, quartz-carbonate stockwork is well developed hangingwall and footwall to the puggy faults. It is this puggy faulting and quartz-carbonate stockwork that hosts the bulk of the sulphide mineralization. Minor higher grade mineralization occurs on the Base Fault where it truncates the anticline hinge.

Moving to the north, the folding becomes more open and the two parasitic folds adopt a more regional north-south trend. This transition occurs at about 12,650mN and corresponds with a plunge reversal in the Trench Syncline. In the area of this reversal, mineralization steps from the anticline-syncline pair to the faulted Trench Syncline (Figure 14-42). This transition has been termed the Culmination Transfer Zone.

14.5.2 Geological Models

Geological modeling undertaken was essentially identical to that described for the Southern Models described above.

14.5.3 Domains

Basic statistics and variographic analysis was completed on the interpreted mineralization wireframes in the O' Dwyer' s-Robbin' s Hill area and also in the Farley' s-Sharkey' s area. Each area was subdivided into two domains chosen based on (in order of decreasing importance) geology, variography and statistics.

In the O' Dwyer' s-Robbin' s Hill area, oxide and sulphide mineralization was grouped into single domains for the Porphyry and the Faults domains because there is very little difference in the statistics of the oxide and sulphide mineralization for the following domains:

In the Farley' s-Sharkey' s area there is sufficient statistical differences between the oxide and combined transitional-fresh zones to isolate these during interpolation of gold grades.

The domains in the Robbins Hill Model area include:

- Porphyries
- OD West Dippers
- RH East
- RH NW Strike
- RH Syncline splay
- Farley' s West Dippers
- Sharkey' s West Dippers
- Sharkey' s East Dippers
- Sharkey' s NE Strike
- Reads-Woolshed

Gold Domains

The Porphyries domain encompasses all quartz-feldspar porphyry hosted gold mineralization at the Robbin' s Hill (RH), O' Dwyer' s (OD) Central, O' Dwyer' s South and Woolshed Pits. The Porphyries domain is low-grade with rare extreme gold grades occurring on the contact.

The OD West Dippers domain encompasses structurally controlled, sediment hosted mineralization associated with north-south striking and variably west dipping faults (50° to 70°) proximal to the existing O' Dwyer's and Robbin's Hill Pits. Farley' s and Sharkey' s West Dipper domains are also defined by mineralization associated with west dipping fault structures proximal to the existing Farley' s and Sharkey' s Pits respectively. The Farley' s Faults domain dip 50°-70° west and show a change in strike from 345° at Farley' s Pit to 360° south of Sharkey' s Pit. The faults are characterized by low grades with rare high-grade values.

The RH East and RH NW strike Domains delineates zones of mineralization concentrated on the eastern side of the Robbins Hill Pit. Domain geometries are variable.
The RH Syncline Splay Domain is located in the north and northeast corner of the Robbins Hill Pit. Mineralization within this domain is associated with the Robbins Hill syncline structure. The domain geometry varies from synformal morphology in the north-eastern side of Robbins Pit to east dipping discordant zones of mineralization cutting through the eastern syncline limb in the northern end of Robbins Hill Pit.
The Sharkey' s East Dipper domain dips 45° east, strikes north and generally has low continuous grades in the fresh domain, but is more erratic in the oxide materials.
The Sharkey' s NE strike domain is a steeply west dipping to sub-vertical domain that extends from the northern end of Robbins Hill Pit to the western flank of the Sharkey' s Pit in the far north of the Mining Lease.
The Reads Woolshed Domain encompasses mineralization associated with west dipping fault structures proximal to the Reads and Woolshed Pits and at depth below the existing Farley' s Pit.

Oxidation Domains

Four 'material' domains were constructed, similar to that described for the Southern Models, although only gold mineralization was interpolated in three of these.
The four domains are:

- Alluvium (near surface transported material, generally barren of gold, largely clay, free digging);
- Oxide (sulphide minerals completely oxidized, Fe-carbonates largely oxidized);
- Transition (sulphide minerals may be partially oxidized, includes zones of mixed fresh and oxide); and
- Fresh (sulphide minerals completely un-oxidized).

14.5.4 Drilling Data

The quality of the drilling is variable in the Robbin' s Hill area. Drilling was conducted from 1989 to 2011, and up until 2001 drilling was focused on oxide heap leach targets and as such cheaper less precise drilling methods were used and dominate the dataset. After 2004, diamond holes were used to aid structural interpretation and often RC pre-collars were diamond tailed.
The model uses more than 1,600 holes of which about 95% are RC holes and 5% are NQ2 and HQ diamond core holes. Drill data was omitted where there was uncertainty of coordinates, dubious down hole surveys and grade or geological mismatch. MineSight drill views were the primary tool used to identify data grade and geological mismatches.

Subsequent to the drill data review, process assay data were:

Imported from the acQuire exploration databases into MineSight using customizable parameter screens; and

Coded for mineralization using 3D gold wireframe solids.

Compositing and Coding

Similar to the Central Area, coded Robbin' s Hill Model area drill data was composited to 2m lengths starting from the point at which the drill hole enters the mineralization envelope. If the final composite was less than 1m it was added to the previous composite making a composite with length between 2m and 3m. Final composites between 1m and 2m in length were left as is. The 2m composite lengths were chosen to reflect the anticipated minimum mining width, to allow across strike variability to be maintained within the data, and because the vast majority of RC drilling samples are 2m long.

Table 14-29 below shows the Robbin' s Hill Model composite statistics.

Table 14-29 Composite statistics by composite length for the Robbin's Hill Model.

Composite Length (m)	Number of Composites	% of Composites	Mean Length (m)	Mean Grade (g/t Au)
<1m	223	2.5	0.01	0.23
≥1and <2	365	4.1	1.11	1.33
≥2	8,380	93.4	2.01	1.3
Total	**8,968**	**100**	**1.91**	**1.27**

Variography

In all domains, the nugget effect (46% to 59%) is typical of gold deposits, but higher than other mineralization domains in the Fosterville Goldfield. Typically low nugget effects elsewhere at Fosterville reflect the fine grained, disseminated nature of the sulphide minerals hosting the elements analyzed and are confirmed by the very low variability exhibited in assay QAQC data. The higher nugget effects modeled for these domains may reflect some mixing of populations, possibly owing to re-mobilization of gold by weathering resulting in erratically distributed extreme gold grades.

The longer range structures in the RH-ODW area possibly reflect high-grade zones occurring where faults intersect the quartz porphyry dykes. The variogram models closely follow the expected geological controls with flat to shallowly south plunging shoots in steeply west dipping faults and sub vertical porphyry contact zones.

14.5.5 Mineral Resource Modeling

Block Models

The most recent Robbin' s Hill Block Model was created in 2011 and has sufficient extents to contain all drilled mineralization beneath the open pits in the area, replicating model extent parameters setup in 2005. Previously, several smaller block models were used to inventory mineralization for the oxide pits in the area.

These models had differing block dimensions and orientations from one another and so combining them into a single unified model was not possible.
The Robbin' s Hill Model has XYZ block dimensions of 4m (EW) by 10m (NS) by 5m (RL). The 4m width was chosen as it is approximates the minimum mining width for both open pit and underground mining. The 10m N-S block dimension is half the section spacing in the most densely drilled areas. The 5m vertical block dimension is the likely mining bench height and allows sufficient resolution for future pit optimization. Block dimensions are identical to those of the Southern and Northern Models described above.
Mineral Resource modeling in the Robbin' s Hill area was undertaken again in 2012. Block model extents encompass all identified zones of gold mineralization in the Robbin' s Hill - O' Dwyer' s Project area.
To facilitate renewed open pit mining in 2012 at O'Dwyer's South a Grade Control (GC) resource model was created with XYZ block dimensions of 2m (EW), 5m (NS) and 5m(RL), with the dimensions chosen to cosmetically better represent likely open pit SMU (Selective Mining Unit) volumes. The block size is identical to those that were previously in use at Harrier and John's open Pits.
In 2012 detailed modeling work was also completed for the Farley's area, for which a Farley's Mineral Resource Model was created. The model has block dimensions identical to those of O'Dwyer's South.
All mineral resource and grade control models in the Robbin's Hill area are shown in Figure 14-43.
The modeling methods and parameters used in the Robbin's Hill Model were largely adopted for use in O'Dwyer's South GC and Farley's Resource Models. Various aspects of the Robbin's Hill Model are described below.

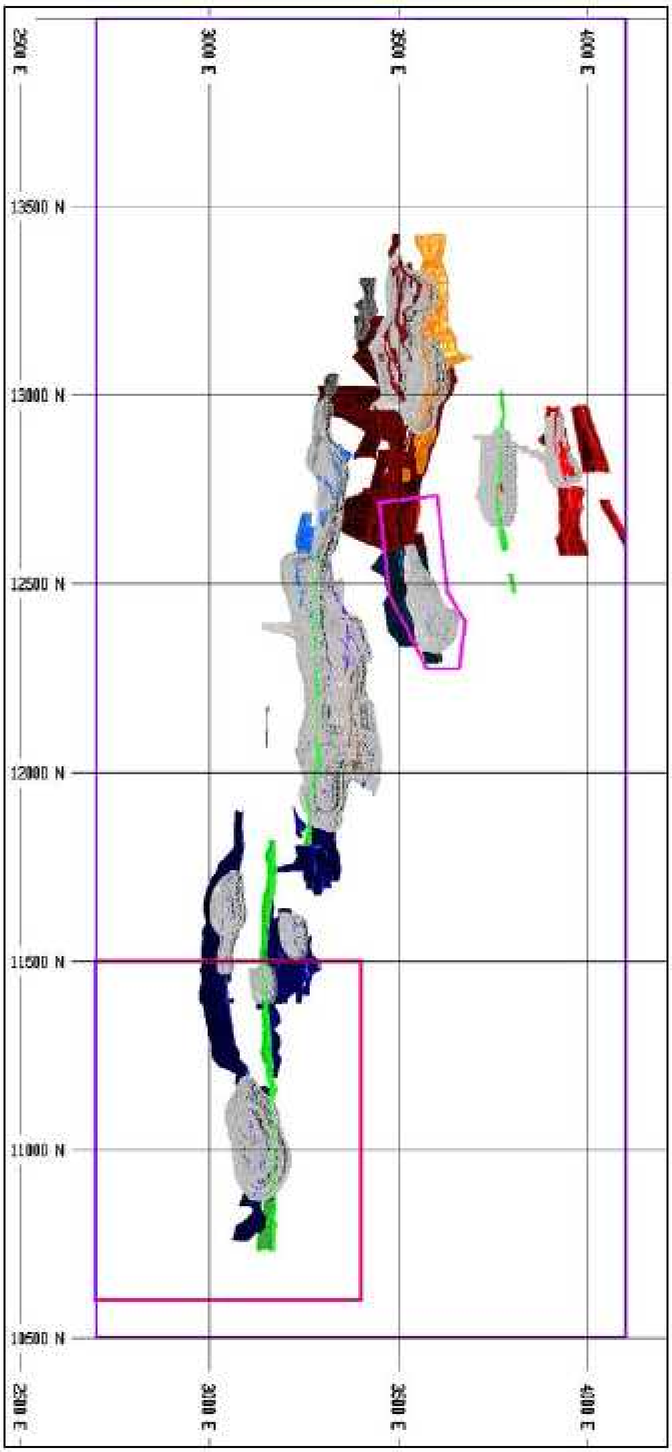

Figure 14-43 Plan view of Mineral Resource and GC Models in Robbin's Hill Area.

Shown is the Robbin's Hill Mineral Resource Model (Purple), covering eastings 2,700mE to 4,100mE, northings 10,500mN to 14,000mN and elevations 4,700mRL to 5,200mRL. More detailed models in the area are used in preference to the Robbin's Hill Mineral Resource Model. These include the O' Dwyer's GC Model (Red) and Farley's Mineral Resource Model (magenta).

Search Criteria

Gold and sulphur grades are only interpolated into blocks meeting the following criteria:

Greater than 1% of the block volume is inside one of the domain envelopes;

Blocks whose search ellipse includes at least 5 composites; and

Blocks whose material code is set to Oxide (3), Transitional (2) or Fresh (1) .

Similarly, only composites meeting the following criteria are used to interpolate any one block:

All composites to a maximum of 20 composites within the search ellipse dimensions and search area limits;

Where more than 20 composites lie within the search ellipse the 20 closest composites in ellipsoid space are used;

Maximum of 10 composites are used from any split-quadrant of the search ellipse; and

The Mineralization and Material values of both the composite and the block must match (i.e. only composites from within the same mineralization envelope and the same oxidation material are used to interpolate a block). However, no distinction is made between Transitional and Fresh materials and both are treated as if they are Fresh.

The search ellipse orientations follow the Kriging axes. The search ellipse dimensions allow the block being interpolated to 'see' two sections along strike and two holes up or down dip.

To check the suitability of the search ellipses used, search ellipses were created in MineSight to allow visual inspection of the composites to be used.

Bulk Density

The bulk density profile established for the Central Area was taken as being appropriate for the Robbins Hill Model area given the similar rock types, levels of oxidation and identical mineralization and gangue mineralogy

Mineral Resource Classification

No mineral resources in the Robbins Hill Area have been categorized as Measured Mineral Resources owing to uncertainties in the quality of the largely historical data used to construct this model.

Two solids were created enclosing regions of geological confidence (Indicated or Inferred Mineral Resources) and these regions were in turn used to code the item RSCAT in the block model. The solids generally enclose areas of approximately equally spaced drilling, but also allow areas where there is reduced confidence in the geological interpretation to be reported to a lower confidence category. The Indicated Mineral Resource solid is always surrounded by a halo of Inferred Resource.

14.5.6 Results

Oxide gold resources exist in the Robbins Hill Model area, notably east of Sharkey' s Pit where exploration drilling in 2007 discovered shallow oxide mineralization. Elsewhere remnant low grade oxide gold mineralization is found below and along strike from previously mined open pits.

Resources in the Farley' s-Sharkey' s area are based on modern face sampling RC methods and substantial diamond drilling and as such the geological information is better than elsewhere in the modeled area.

15 MINERAL RESERVE ESTIMATES

The current Mineral Reserve estimate, from the available Mineral Resource estimates, is presented below in Table 15-1.
CIL Residue Mineral Reserves are distinguished from insitu Mineral Reserves in Table 14-1 on the basis of differing recovery assumptions.

Table 15-1 Mineral Reserves for FGM as at December 31, 2015.

Classification	Proven			Probable			Total		
	Tonnes (kt)	Grade (g/t Au)	In situ Gold (kOz)	Tonnes (kt)	Grade (g/t Au)	In situ Gold (kOz)	Tonnes (kt)	Grade (g/t Au)	In situ Gold (kOz)
Underground									
Central	72	5.11	12	149	4.58	22	220	4.75	34
Phoenix	120	5.24	20	531	8.20	140	651	7.65	160
Eagle	24	8.24	6	149	8.00	38	173	8.03	45
Harrier	17	3.69	2	30	3.32	3	47	3.45	5
Surface									
	0	0.00	0	0	0.00	0	0	0.00	0
Total	232	5.39	40	859	7.36	203	1,091	6.95	244

Classification	Proven			Probable			Total		
	Tonnes (kt)	Grade (g/t Au)	In situ Gold (kOz)	Tonnes (kt)	Grade (g/t Au)	In situ Gold (kOz)	Tonnes (kt)	Grade (g/t Au)	In situ Gold (kOz)
Residues									
CIL Residues	571	7.83	144	0	0.00	0	571	7.83	144
Total	571	7.83	144	0	0.00	0	571	7.83	144

Notes:

For the Mineral Reserves estimate, the Qualified Person is Ion Hann . The Mineral Reserve estimate used a gold price of AUD$1,450 per ounce. The lower cut-off grades applied ranged from 1.6 g/t to 2.7 g/t Au for underground sulphide ore depending upon width, mining method and ground conditions.

Dilution of 20% and mining recovery of 80% were applied to stopes within the Mineral Reserves estimate.

Mineral Reserves are rounded to 1,000 tonnes, 0.01 g/t Au and 1,000 ounces. Minor discrepancies in summation may occur due to rounding.

CIL residues are stated as contained ounces - 25% recovery is expected. Recoveries are based on laboratory and processing plant test work and operating experience.

15.1 Mineral Reserve Estimate

The following sections outline the process undertaken to produce Mineral Reserve estimates from the available Mineral Resource. This section contains descriptions of mineral reserve design parameters, recovery and unplanned dilution factors, cut-off grades and depletion for mined material. This section of the report has been reviewed by Ion Hann who is the Qualified Person for the Mineral Reserve estimate (details in Section 28.1) .

15.1.1 Mineral Reserve Design

The initial stage of the mineral reserve estimation process was the revision of the Mining Method Selection chart. The mining methods that were considered for the mineral reserve estimation process were sill driving, up-hole open stoping, up hole stoping with fill, underhand open stoping with chain and rib pillars and transverse open stoping. These methods were selected based upon previous experience at the Fosterville mine or because they were considered suitable for the ore zone geometry and geotechnical conditions present and expected.

15.1.2 Open Stope Design and Reserve Parameters

Stope reserve shapes were created to cover all active and planned mining areas. These stope shapes did not necessarily reflect the final stope strike and/or crown pillar dimensions. Stoping widths vary from 3m out to 10m. Mining method selection criteria and applied design parameters are described in the Mining Methods Selection process (see Figure 15-2).

The open stope reserve wireframe design parameters applied were:

- Strike length dictated by grade distribution in block model
- Minimum true bench width of 3m
- Maximum benching height of 20m vertical from backs to floor
- Internal waste incorporated within the stope block design

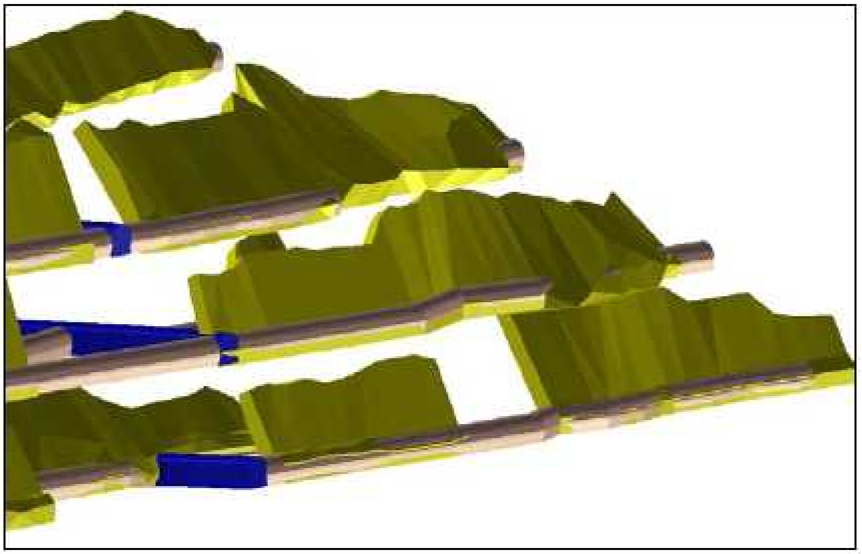

Figure 15-1 An example of an open stope reserve wireframe design.

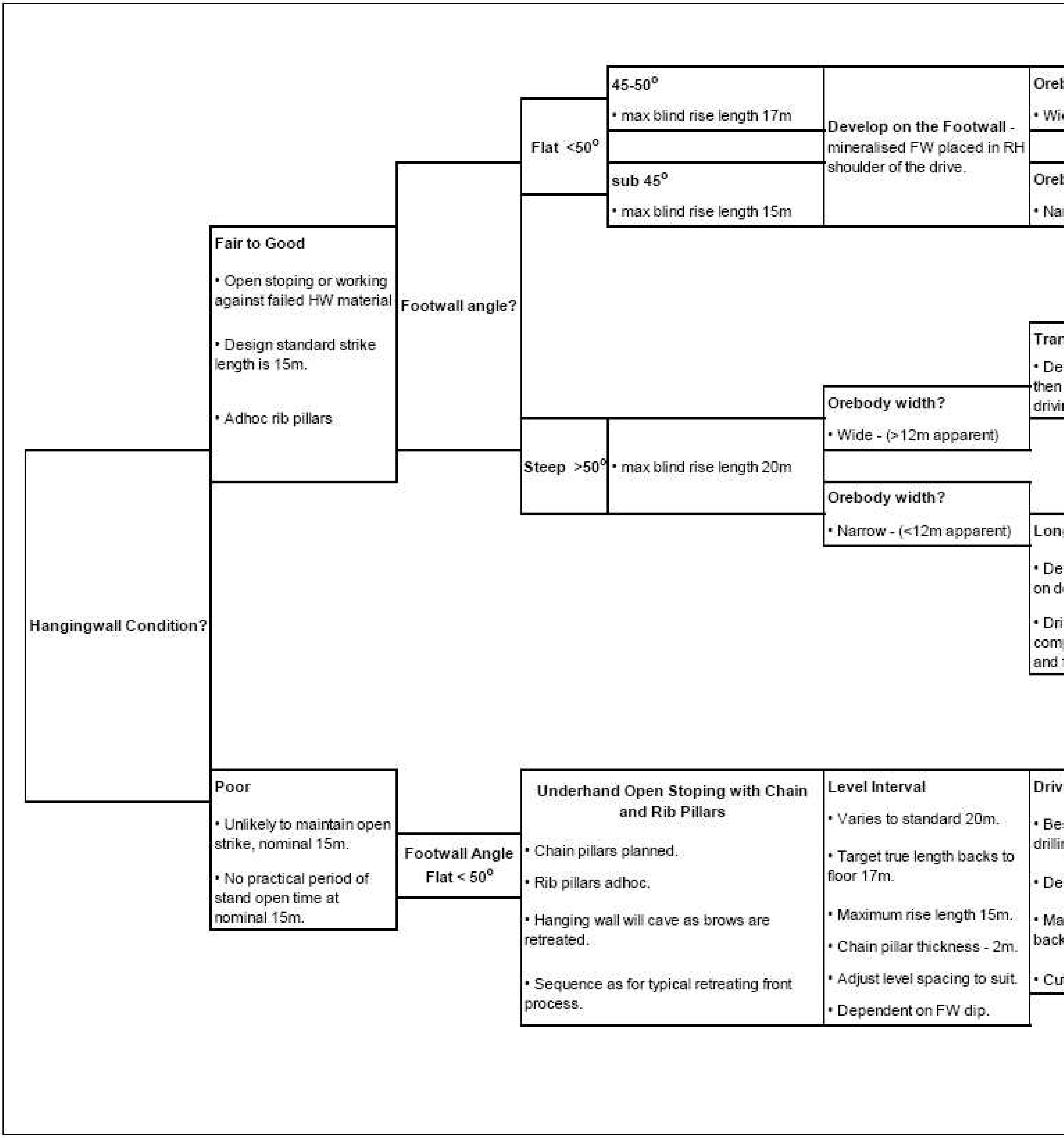

Figure15-2Miningmethodselection.

Mining recovery from open stopes at Fosterville is principally influenced by the following factors:

- Accuracy of the geological interpretation.
- Accuracy of the production hole drilling.
- Stope dimensions.
- Sill drive dimensions and position relative to bench stope.
- Presence or absence of adjacent filled voids and pillars.
- Geotechnical integrity of stope and sill drive walls.

The above factors manifest themselves as ore loss in the following ways:

- The need for planned pillars due to accessing of ore blocks (i.e. top down mining sequence).
- Frozen rings due to ground movement.
- Bridged stopes.
- Failure of the stope to break back to a main structural plane of weakness.
- Unplanned ore pillars left to improve ground support.

Unplanned dilution in open stopes at Fosterville is a function of the following factors:

- Regional geotechnical conditions.
- Location of sill drives relative to the open stope.
- Width of sill drives relative to the open stope width.
- Production drilling accuracy.
- Quantity, quality and type of ground support in sill drive walls.
- Speed of ore extraction from active stopes.
- Length of time sill drives have been open before stoping commences.

In order to correctly apply recovery and dilution factors to all stopes in the Mineral Reserve, factors such as orebody dip, rock RQD and development and stope sequence were considered.

Table 15-2 and Figure 15-3 show the recovery and dilution factors that were applied to the reserve blocks:

Table 15-2 Recovery and dilution factors for the reserve blocks as displayed in Figure 15-3

Description	Recovery Factor - Tonnes	Dilution Factor - Tonnes	Comments
Stoping - Phoenix	80%	20%	Top down, crown and rib pillars, and/or CRF, underhand open stoping with chain and rib pillars
Stoping - Central	80%	20%	Top down, crown and rib pillars, and/or CRF, underhand open stoping with chain and rib pillars
Stoping - Harrier/Osprey	80%	20%	Top down, crown and rib pillars, and/or CRF, underhand open stoping with chain and rib pillars
Stoping - Robin	80%	20%	Open stope
Strike Development	100%	15%	

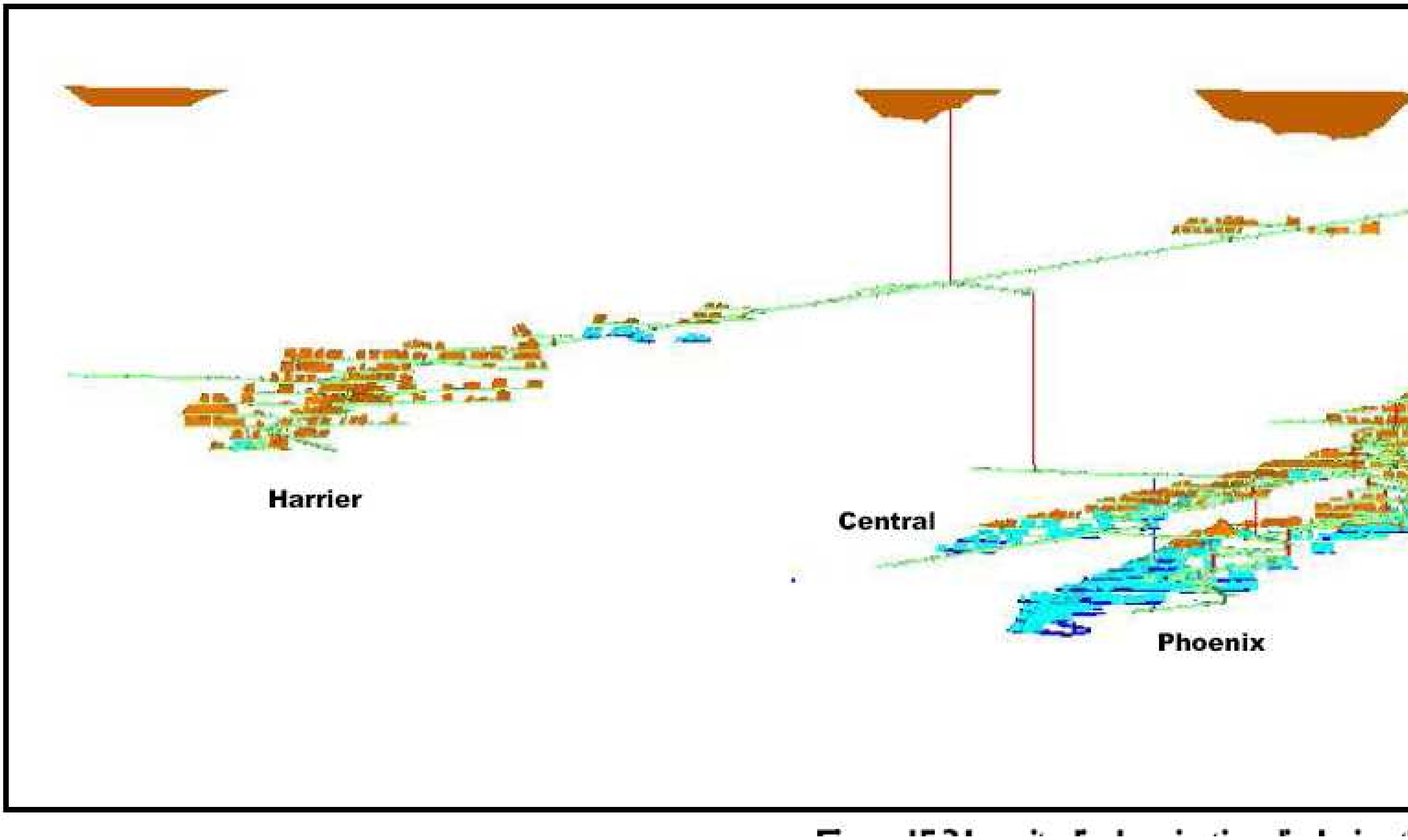

Figure15-3 Longitudinal projection displaying the mining blocks referred to inTable15-2.

Gold Lower Cut-Off Grades

Table 15-3 shows the calculated lower cut-off grades used in the estimation of the Mineral Reserve. Cost assumptions are based on the 2016 budget and 2015 full year performance.

Table 15-3 Gold lower cut-off grades.

Description	g/t Au
Open Stope - full	2.71
Open Stope - marginal	1.58
Development - marginal	3.34

For certain other situations, a lower cut-off grade is applied. For development which is justified for other reasons (i.e. access to a higher grade block or infrastructure considerations), the cut-off grade is lowered to reflect that the material only has to cover the non-mining costs to break even. This is only applied if the development material had to be trucked to surface anyway and that it is not displacing higher-grade ore from the mill. Likewise for incremental stoping production where the development has already been mined (i.e. for access to a higher-grade block), the cut-off grade is lowered to reflect that the development cost has already been incurred.

Stope and development shapes are limited in their extremity by the application of appropriate COGs (Table 15-3) and a full conceptual design is subsequently created around the resultant shapes. This design includes but is not necessarily limited to; decline design, associated level infrastructure and vertical development.

Physicals generated from the design are applied against budget costs and assumptions to provide an economic model by level and area (Table 15-4). This model is capable of representing various cost structures and is utilized as the final hurdle point for determination of inclusion/exclusion of material into the mine plan and reserve statement.

Table15-4:DevelopmentCostsandphysicalsspreadsheet.

Level Economic Evaluations 2016 Reserves	RL: Drive: Area: Model: Orebody:	4220 LEVEL PHOENIX 1512_PRM
Dec / Inc (m)		-
Cap Other (m)		279
Cap RAR (vertical m) - big raise bore		-
Cap Escapeway - RaiseBore & Safescape		-
Cap RAR (vertical m) - D&B not supported		-
Cap RAR & E/way (vertical m) D&B Supported		-
Waste Operating (m)		140
Ore driving (m)		328
Ore driving (tonnes)		22,530
Ore driving (ounces)		4,735
Stope (tonnes)		88,212
Stope (ounces)		25,164
Production Drilling (m)	3 tonnes/drill m	29,404
CRF (tonnes)		125,000
RF (tonnes)		
Check Ore (m)		69
Dilution Factor - Development Material	100%	1.00
Dilution Factor - Stope Material	100%	1.00
Recovery Factor - Development Material	100%	1.00
Recovery Factor - Stope Material	100%	1.00
Indic Prod ore (t)		88,212
Indic Prod ore (g/t)		8.87
Indic Prod ore (oz)		25,164
Indic Dev ore (t)		22,530
Indic Dev ore (g/t)		6.54
Indic Dev ore (oz)		4,735
Total Prod ore (t)		88,212
Total Prod ore (g/t)		8.87
Total Prod ore (oz)		25,164
Total Dev ore (t)		22,530
Total Dev ore (g/t)		6.54
Total Dev ore (oz)		4,735
Total ore (t)		110,741
Total ore (g/t)		8.40
Total ore (oz)		29,899

CAPITAL		
____________**Access**	**Unit Cost**	
Decline Dev.	$7,613/m	
Other Dev.	$6,341/m	
Equipment		
_________**Sustaining**		
Decline Dev.	$7,613/m	$ -
Other Dev.	$6,341/m	$ 1,769,139
Vent rise	$4,160/m	$ -
RAR & Escape way	$7,685/m	$ -
OPERATING		
Development		
Ore	$6,664/m	$ 2,182,874
Waste	$6,664/m	$ 932,960
Production		
Ground Support (Stope)	$5.47/stope tonne	$ 482,517
Drilling	$31.85/drill m	$ 936,513
Blasting	$26.10/charge m	$ 575,580
Load & Truck	$3.31/tkm	$ 1,864,002
Backfill - CRF	$15.00/placed t	$ 1,875,000
Backfill - RF	$5.33/placed t	$ -
Other Fixed		
Mine Administration - includes geology	$5.97/t mined	$ 661,126
Milling Administration	$24.01/t mined	$ 2,658,900
Finance & Administration	$15.93/t mined	$ 1,764,110
Site Capital Sustaining	$30.79/t mined	$ 3,409,727
Other Variable		
Mine General	$2.95/t mined	$ 326,687
Milling	$18.23/t mined	$ 2,018,815

15.1.3 Depletion and Results

The Mineral Reserves reported above are largely the result of work based on data to December 31st 2015 and reported by Newmarket Gold under Canadian reporting requirements in accordance with NI43-101. The evaluation models have been depleted for material mined up to December 31st 2015. The process involved the generation of surveyed solid models for the mined development and stope areas and then running a depletion process in order that the depleted areas be excluded from the mineral reserve.

Results for the Mineral Resource and Mineral Reserves contained in the Fosterville operating areas are provided in Table 14-1 and Table 15-1, respectively.

Infrastructure required for the exploitation of the stated reserves are either in place or have been planned to be developed within the LOM plan generated through the reserving process. All works fall within the granted mining lease boundaries and are covered within the existing approved work plan. It is unlikely that either infrastructure or permitting could materially affect the stated reserve position.

There are no known political, legal, environmental or other risks that could materially affect the potential development of the mineral reserves.

16 MINING METHODS

Since the completion of the Harrier Open Cut Mine in early December 2007, the sole source of ore had been the underground operations until Q2 2011 when ore feed became available from a series of open pit cut backs on the Harrier Pit, John’s Pit and O'Dwyer's South Pit. Since the completion of O'Dwyer's South Cut back in Q4 2012, the sole source of ore has been from the underground operations. The current Life of Mine (LOM) plan contains ore sourced from underground operations only (Figure 16-1).

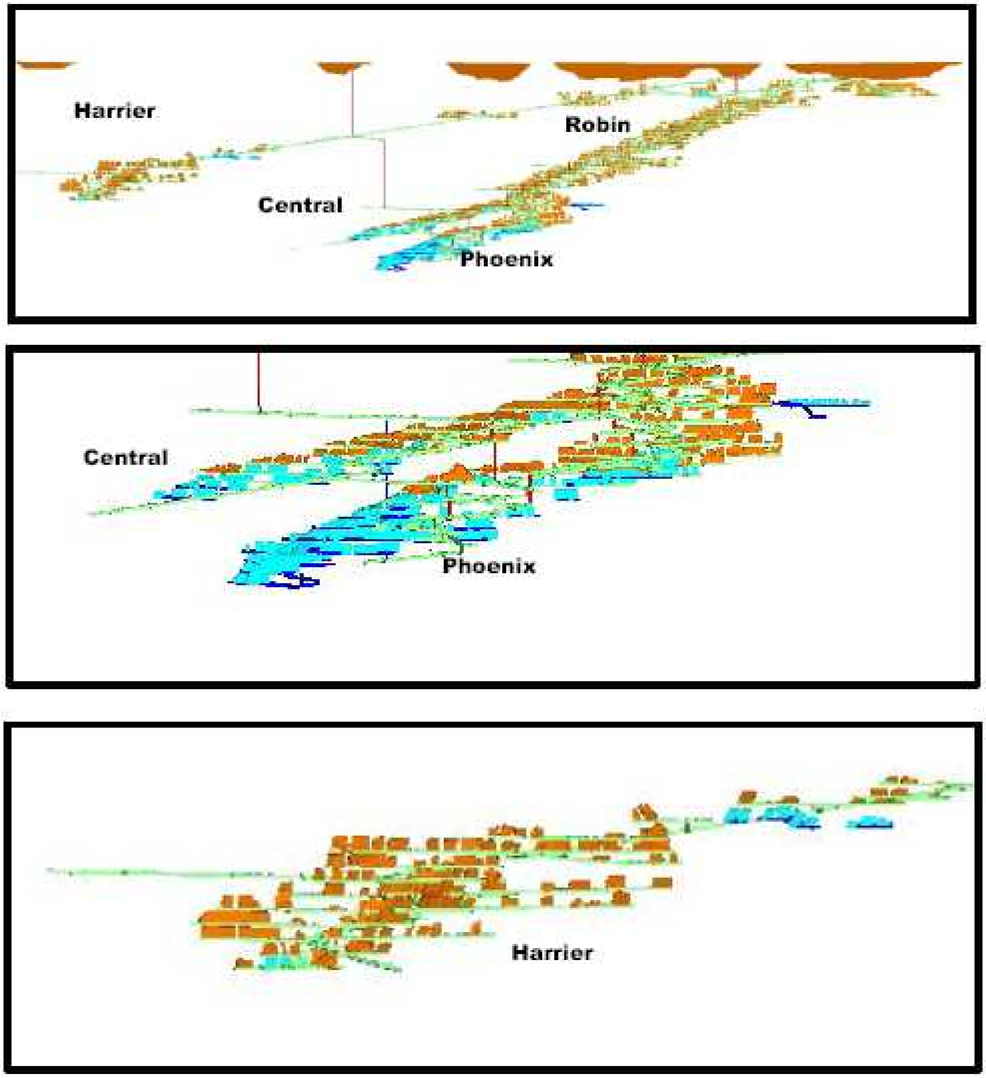

Figure 16-1 Longitudinal projection of actual and proposed mining layout as at January 1, 2015.

The underground mine commenced declining in March 2006 with production first recorded in September 2006. Development and stoping have been conducted in the Phoenix, Falcon, Ellesmere, Kink, Vulture, Raven, Robin and Harrier Orebodies since that time. As at January 1, 2016 works are planned to continue in the Phoenix, Central, Harrier and Robin (Robin material is included with Central material in the reserve table (Table 15-1)) Orebodies. All areas are planned to be extracted using open stoping techniques with the application of CRF where applicable and practical. Selection of the specific mining method within the open stoping regime is based upon previous experience at the Fosterville mine and expectations of ore zone geometry and geotechnical conditions (Figure 15-2) A standard level interval of 20 vertical meters can be applied across all mining areas. However, this can be varied as is required to maximize the extraction of the economic material. The Phoenix to 4240rl, Central and Robin Orebodies are accessed from a footwall decline position while the Phoenix below 4240rl and Harrier Orebody accessed from the hangingwall.

Underground mining is conducted using a conventional fleet including jumbos, production drills, loaders, trucks and ancillary equipmen. Current mining is undertaken as owner miner.

The processing path for the ore involves crushing and grinding followed by flotation, bacterial oxidation and CIL circuits. The bacterial oxidation process uses BIOX technology, operated under licence from Goldfields Pty Ltd. The flow sheet can be seen in Figure 16-2. The processing capacity is approximately 830ktpa.

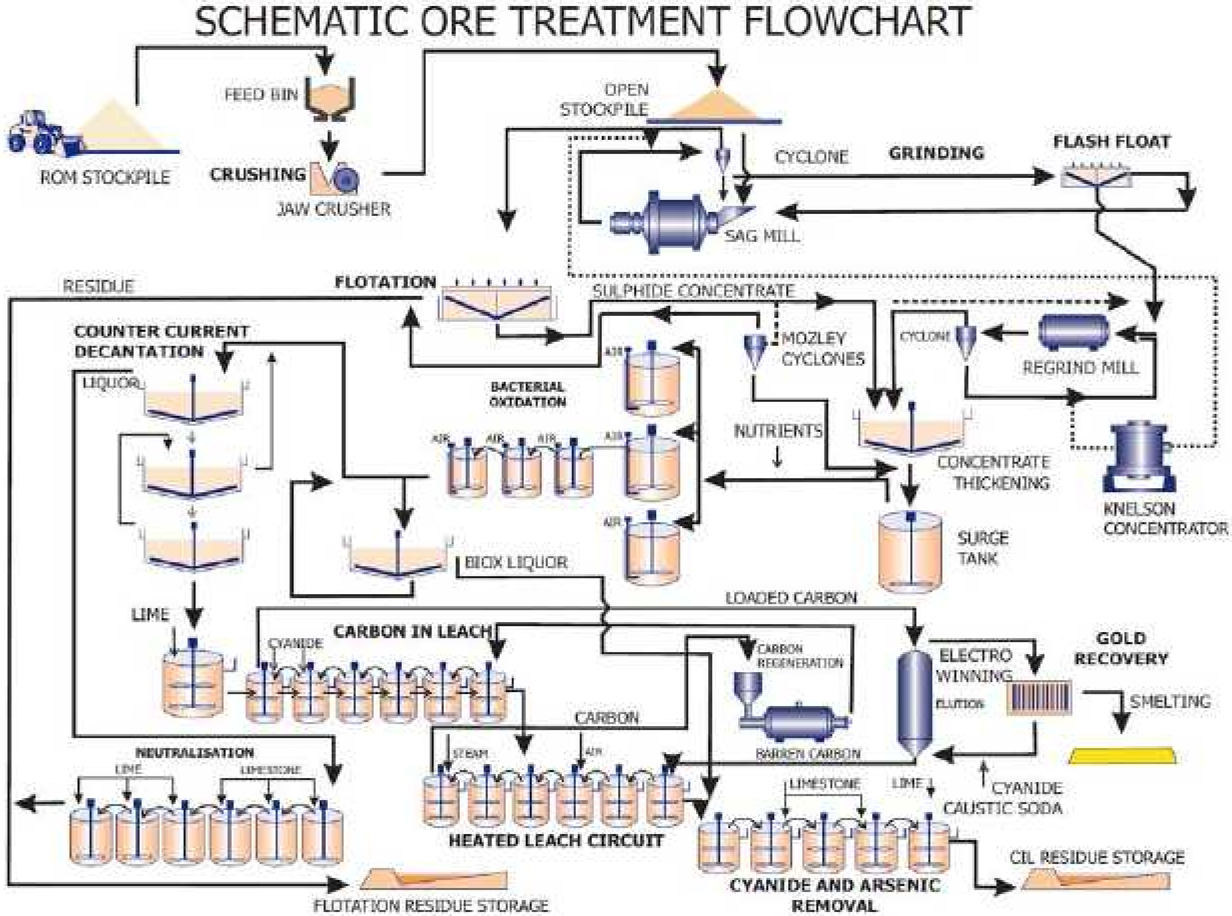

Figure 16-2 Schematic ore treatment flow sheet.

The production forecast contained in Table 16-1 forms part of the latest Life-of-Mine (LOM) Model. This LOM Model was for the period 2016 (FY-16) to 2021 (FY-21).

Table 16-1 Production forecast for the years 2016-2021 (Current LOM plan).

Fosterville Gold Mine	LOM Model						
	LOM	FY-16	FY-17	FY-18	FY-19	FY-20	FY-21
Total ore mined (t)	3,109,741	659,741	650,000	650,000	650,000	500,000	0
Ore Milled (t)	3,119,742	669,742	650,000	650,000	650,000	500,000	0
Ore Milled Head Grade (g/t)	5.80	5.78	5.80	5.80	5.80	5.80	0
Recovery (%)	89.0	89.0	89.0	89.0	89.0	89.0	0
Gold Produced (Oz)	517,338	110,731	107,875	107,875	107,875	82,981	0
Tails retreat (Oz)	37,000	1,200	1,200	1,200	1,200	1,200	31,000
Total Gold Produced (Oz)	554,338	111,931	109,075	109,075	109,075	84,181	31,000

17 RECOVERY METHODS

Since the commissioning of the processing plant in 2004, all processing models for the mill have been based on actual plant performances. The processing budget takes into consideration the mining schedule (ore source location, tonnes to be mined and gold grade), and predicted sulphur grades to be processed. Recovery data for Fosterville is detailed in Table 17-1.

Section 13 in this report describes the metallurgical test-work that was conducted during the feasibility stage.

Table 17-1 Actual plant performances (2008-2014).

Plant Parameter		2008	2009	2010	2011	2012	2013	2014	2015
Tonnes Milled	t	540,725	781,878	817,535	785,503	786,572	792,166	814,835	703,788
Sulphur Feed grade	%	1.64	1.71	1.6	1.59	1.44	1.35	1.36	1.34
Feed Grade	g/t Au	5.42	4.79	4.57	4.87	4.36	4.53	4.62	6.11
Flotation recovery	%	92.4	96.2	96.2	96.7	95.0	95.9	95.7	96.6
BIOX recovery	%	97.8	99.0	98.7	98.4	97.8	98.0	98.6	98.5
Sulphide Oxidation	%	97.5	96.3	98.6	97.7	97.7	98.2	98.1	98.3
CIL recovery	%	78.7	86.2	79.8	81.3	80.5	86.2	87.1	90.9
Heated leach recovery	%	0.0	0.3	7.1	6.0	7.6	4.5	4.6	2.0
Overall Leach recovery	%	78.7	86.6	86.9	87.3	88.1	90.7	91.6	92.9
Overall Plant recovery	%	71.2	85.0	82.5	83.0	82.0	85.2	86.5	88.5
Mining Au produced	troy oz	66,984	102,336	99,032	102,048	90,358	98,354	104,518	122,362
Retreat: Leach tails: tonnes	t	0	9,634	13,222	4,495	2,623	854	4,951	4,519
Retreat: Leach tails: grade	g/t Au	0	10.25	10.37	8.27	6.98	7.05	10.48	10.75
Retreat: Leach tails: recovery	%	0	32.5	30.3	12.2	12.1	35.2	49.0	46.3
Retreat: Leach tails: Au produced	troy oz	0	1024	1,410	154	80	69	824	734
Total gold produced	troy oz	66,984	103,360	100,442	102,201	90,439	98,423	105,342	123,096

The process plant incorporates the following unit operations:

- Single stage crushing with a primary jaw crusher.
- Open stockpile with reclaim tunnel.
- Semi autogenous grinding (SAG) mill.
- Flotation circuit to produce a gold bearing sulphide mineral concentrate and a barren residue. The flotation concentrate is then thickened.
- Bio-oxidation circuit consisting of BIOX® reactors to oxidize the flotation concentrate, releasing gold from the sulphide mineral matrix.
- A three-stage CCD circuit to separate the gold bearing oxidized solid residue from the solubilized acid oxidation products.
- A liquor neutralization circuit to neutralize acid and precipitate arsenic as stable basic ferric arsenate and sulphate as calcium sulphate (gypsum) using both ground limestone and lime slurries.
- A limestone grinding facility comprising a single wet ball mill operated in closed circuit with hydrocyclones to produce ground limestone slurry for neutralization of sulphuric and arsenic acids produced from oxidation of gold bearing sulphide minerals.
- Carbon-in-leach (CIL) circuit, with a pH adjustment tank at the head of the circuit, to leach gold from oxidized material and load the cyanide soluble gold onto activated carbon.
- Heated Leach (HL) circuit to combat preg- robbing capabilities of the non-carbonaceous carbon frequently present in the Mill feed. Specialized in-house technology unique to Fosterville.
- Pressure Zadra elution circuit to remove gold from carbon, followed by recovery by electrowinning and smelting to doré.

A schematic flowsheet detailing unit operations is presented in Figure 16-2

The plant was laid out on either side of a central rack in order to facilitate the distribution of reagents, services, and piping arrays. Individual plant areas are separately bunded to isolate and contain spillage. Storm water and abnormal spillage events report to an existing drainage channel, which discharges to a separate containment dam.

The layout of the comminution circuit allows for installation of a pebble crushing circuit should it be required, and a secondary ball mill to increase grinding circuit capacity. Space was left in the area layouts for additional tank farms and equipment to accommodate a nominal increase in plant capacity. Space exists to the east of the plant site to duplicate existing facilities to double plant throughput if required.

Plant commissioning began in November 2004 with first gold production in Q1 of 2005.

Crushing and Milling

The crushing circuit has the capacity to operate 24 hours per day, 7 days/week, at the design availability of 80%.

Run of Mine (ROM) ore is reclaimed from stockpiles on the ROM pad and fed to a bin by front-end loader, blending the ore in the process. Ore is then fed to a 760 mm x 1,372 mm single toggle jaw crusher by a vibrating grizzly feeder and minus 100 mm crushed ore is conveyed to a coarse ore open stockpile with reclaim tunnel providing feed to a semi autogenous grinding (SAG) mill.

Dust suppression measures are installed at the ROM bin. The crusher discharge and conveyor transfer points both being fitted with dust collectors.

Crushed ore is fed at a controlled rate onto a conveyor feeding 3,500 kW SAG mill (~6.1m in diameter x 6.1m) . The ore is ground to a P80 of 75 µm in closed circuit with hydrocyclones to liberate sulphide minerals containing gold from the barren gangue minerals. The milling circuit operates 24 hours per day with a throughput of up to 120 dry tph.

Flotation

Hydrocyclone overflow from the SAG mill gravitates to the flotation circuit where the gold containing sulphide minerals are concentrated into a flotation concentrate containing about 8% to 10% of the feed mass with a barren flotation residue which is rejected from the process.

The design basis for the flotation circuit is to maximize gold recovery to a concentrate grading approximately 20% S. The flotation circuit consists of a rougher-scavenger circuit to produce a single, combined concentrate fed to a cleaner circuit. A single stage of cleaning was included to upgrade lower grade scavenger concentrates from lower grade feed to maintain constant sulphur to the BIOX® circuit. Cleaner tailing is recycled to the head of the rougher circuit.

Flotation reagents such as the following are added to the hydrocyclone overflow launder:

Activator - copper sulphate
Collector - potassium amyl xanthate (PAX)
Promoter – Butyl dithiophosphate
Frother

Reagent selectivity is a key aspect of the flotation circuit management, based not just on performance, but also toxicity aspects to the downstream Bacterial Oxidation circuit.

Flotation residue gravitates to a tailings hopper and then is pumped to the flotation residue storage facility together with the products from neutralization of the BIOX® liquor.

Flotation concentrate is thickened in a high-rate thickener prior to feeding the BIOX® circuit.

Oxidation - BIOX®

Due to the different design availabilities between the milling/flotation circuits and BIOX® circuit, and the need for steady operation of the BIOX® circuit, a surge tank with a live capacity of about 48 hours acts as a buffer between the circuits.

The BIOX® bacteria are sensitive to chloride levels in the water, and management of BIOX® feed dilution water quality to <1000ppm Cl^- is critical for the health of the BIOX® circuit. Likewise, cyanide and thiocyanate species are also toxic materials to the bacteria, hence the Flotation and Neutralization waters, plus CIL decant liquors are managed separately at the Fosterville operations to eliminate any processing risks.

Nutrient solution is dosed to the feed splitter box to maintain the correct levels of nitrogen (N), potassium (K) and phosphorous (P) levels in the BIOX® reactors.

The BIOX® culture is kept active in the reactors by controlling the slurry conditions within specific ranges. The oxidation reactions are exothermic and it is necessary to constantly cool the slurry. The reactors are equipped with cooling coil baffles through which cooling water is circulated to control the slurry temperature at about 43º C in each reactor.

Oxygen requirements for sulphide oxidation are significant and medium pressure air is injected into each of the reactors.

The slurry pH in each of the reactors is controlled between 1.0 and 1.6 by addition of ground limestone. Hence the corrosive nature of the BIOX® slurry and the potential risk for elevated chloride levels resulted in selection of SAF 2205 stainless steel for equipment in the BIOX®, CCD, and neutralization circuits.

The oxidized product discharged from the final secondary BIOX® reactor gravitates to a product hopper from where it is pumped to the first of three CCD thickeners.

During bio-oxidation iron, sulphur and arsenic is solubilized and is washed from the solid oxidized gold containing residue in the series of three CCD thickeners. A three-stage CCD circuit with a wash ratio of 4.0 is used to ensure cyanicides, soluble arsenic and acid is reduced to levels acceptable in the oxidized concentrate prior to the CIL process. Process water is used as wash water in the CCD circuit and is added to the feed tank ahead of the third (last) CCD thickener. The underflow from the last CCD thickener (washed product) is pumped to an agitated pH adjustment tank ahead of the CIL circuit.

The acidic solution overflowing the first CCD thickener is pumped to the first of 6 agitated neutralization tanks in series and the solution flows from tank to tank via launders. By-pass launders allow tanks to be taken off line for cleaning and maintenance. In the neutralization circuit the majority of the sulphuric acid is neutralized and precipitated as calcium sulphate (gypsum) and the soluble arsenic and iron precipitated as stable basic ferric arsenate.

The neutralized effluent gravitates to a flotation residue hopper and is pumped with flotation residue to the residue storage facility.

Mozley Cyclones

Ahead of the BIOX® surge tank, the Mozley de-sliming cyclones were installed in April 2008. The Mozley cyclones are used when the feed blend to the flotation circuit is more than 0.3% NCC. The rougher and cleaner concentrate from the flotation concentrate is run through the Mozley cyclones

The cyclone clusters come in two sets of 20 cyclones and have a typical spigot /vortex finder arrangement of 2.2/7.0mm. The cyclones are fed at a pressure of 300Kpa resulting in typical mass split of 60% to the underflow. Typical feed rate of 40 – 50 m^3/hr @ 16% solids with 30 – 40m3/hr at 5 - 8% solids reporting to the overflow tailings.

Leaching

Six adsorption tanks are identical in size at 190 m^3 with a total circuit residence time of about 48 hours at a 30% pulp density. Test-work indicates that the leaching of the oxidized residue plateaus at 36 to 48 hours. Underflow from the last CCD thickener is pumped to the pH adjustment tank and lime slurry is used to neutralize residual acid and raise the pH of the pulp to 11.

Carbon concentrations (20 g/L – 30 g/L) are maintained in all tanks to ensure high gold adsorption efficiency and achieve a low soluble tail. The last CIL tank can be used for tails retreat storage.

Heated leach

CIL discharge is fed to heated leach circuit, which was commissioned in April 2009. The process utilizes heat from steam injection and caustic to facilitate gold release from native carbon.

The heated leach circuit consists of 6 by 75 m^3 tanks with a residence time of 8 to 12 hours. The first three tanks are heated. The last three tanks are cooled to avoid loss of gold in solution. The heated leach process is effective in destroying WAD cyanide to < 1 ppm and has replaced the former detoxification circuit.

Elution and Gold Electro-winning

The following operations are carried out in the elution and gold room areas:

- Acid washing of carbon;
- Stripping of gold from loaded carbon using a pressure Zadra elution circuit;
- Electro-winning of gold from pregnant solution; and
- Smelting of electro-winning product.

The elution and gold room areas operate up to 7 days per week, with the loaded carbon recovery on nightshift and the majority of the elution occurring during dayshift. The 3.5t pressure Zadra elution circuit consists of separate rubber lined acid wash and stainless steel elution columns.

Energy, water and major process reagents consumed by the processing plant are all readily available in Australia. There is not expected to be any significant increases or decreases to the current consumption rates.

18 PROJECT INFRASTRUCTURE

All project infrastructures are in place servicing mining and processing operations (Figure 18-1).

18.1 Surface Infrastructure

18.1.1 Plant

The process plant site was selected close to the western boundary of the Fosterville Mine Lease, as it

- Offers easy access from the existing public road system
- Minimizes haulage distances from mining operations, particularly, the underground portal location
- Minimizes the potential for noise impact on nearby residential areas to the east and south by allowing waste dumps and noise abatement bunds to be constructed to the east of the plant site.

The process plant has a nominal capacity of 830,000 tpa and incorporates the following unit process operations (Figure 18-2):

- Single stage crushing with a primary jaw crusher
- Open stockpile with reclaim tunnel
- Semi autogenous grinding (SAG) mill.
- Floatation circuit to produce a gold bearing sulphide mineral concentrate and a discardable barren residue. The flotation concentrate is then thickened.
- A bank of de-sliming hydrocyclones for removing native carbon from flotation concentrate
- Bio-oxidation circuit consisting of BIOX® reactors to oxidize the floatation concentrate, releasing gold from the sulphide mineral matrix
- A three stage CCD circuit to separate the gold bearing oxidized solid residue from solubilized acid oxidation products.
- A liquor neutralization circuit to neutralize acid and precipitate arsenic as stable basic ferric arsenate and sulphate as calcium sulphate (gypsum) using both ground limestone and lime slurries.
- A limestone grinding facility comprising a single wet ball mill operated in closed circuit with hydrocyclones to produce ground limestone slurry for neutralization.
- Carbon-in-leach (CIL) circuit, with a pH adjustment tank at the head of the circuit, to leach gold from oxidized material and load the cyanide soluble gold onto activated carbon.
- A heated leach circuit consisting of 6 x 75 m^3 tanks to recover 'preg-robbed' gold from native carbon.
- Pressure Zadra elution circuit to remove gold from carbon, followed by recovery by electrowinning and smelting to dore.

The plant is laid out on either side of a central rack in order to facilitate the distribution of reagents, services and inter-area piping. Individual plant areas are separately bunded to isolate and contain spillage. Storm water and abnormal spillage events report to an existing drainage channel, to the west of the plant area, which discharges to an existing containment dam to the north.

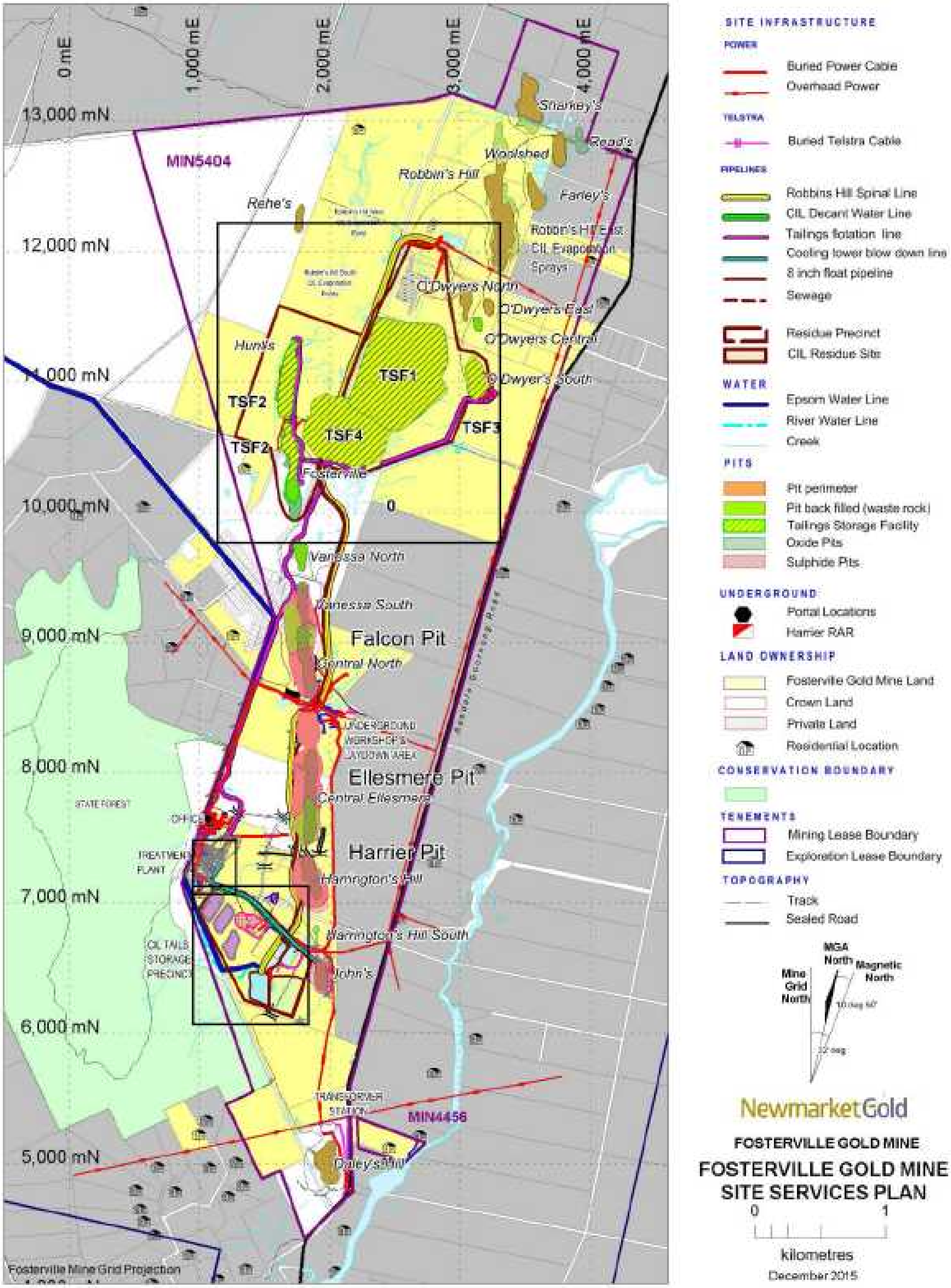

Figure 18-1 Fosterville Gold Mine Site Services Plan.

Figure 18-2 Fosterville Processing Plant Area Plan.

18.1.2 Buildings

The site buildings comprise of administration, processing and mining office complexes, toilet/shower/change room facilities, store/ warehouse, light vehicle and heavy vehicle workshops, a surface maintenance workshop and core shed facility.

The site is serviced by security infrastructure, phone and internet services.

18.1.3 Power

Site power is supplied by the Fosterville Terminal Station (FVTS) which is a zone substation on the 220kV power line from Bendigo to Shepparton (BETS-SHTS). The terminal station is owned by Fosterville, operated by SP Ausnet and maintained by Powercor.

The terminal station has a single 15/20MVA ONAN/ONAF 220/11kV transformer.

An overhead 11kV power line runs from the FVTS to the processing plant. The power line is 2,800m long at consists of 19 poles.

At pole 9 there is an 11kV switch room which supplies the U/G operation.

The processing plant has five 11kV/ 415V transformers and low voltage MCC' s to supply and control the processing plant.

There is also an 11kV 3,500kW SAG Mill motor and three 11kV 750KW motors for the BIOX Blowers.

The processing plant also has a Power Factor Correction unit.

Power consumption in the processing plant is approximately 7,000kw at a power factor of 0.98.

There are also a couple of 22kV supplies into site, which supply remote areas for site water management as well as the main administration offices.

The site also has a 2.5km long 11kV cable from the U/G settling dams to the in pit Tails MCC which has a 750KVA 11kv/415V transformer.

18.1.4 Tailings

There are two separate residue streams at Fosterville, a flotation /neutralization residue (Figure 18-3) and a cyanide bearing residue (Figure 18-4):

> The flotation / neutralization residue is a combination of flotation tails (95%) which is ground ore and neutralized liquor containing precipitated solids (5%) from the oxidation process. These tailings are either stored within an above ground paddock style Residue storage facility, or within an In- Pit facility. Fosterville operates Victoria' s first In-Pit facilities, whereby through extensive hydro- geological modeling, abandoned oxide ore pits where identified as preferred storage options. In-Pit facilities offer significantly lower capital and operating costs compared to above ground facilities, and also contribute to the overall rehabilitation of the mine site. Water from these facilities is reused back through the milling, flotation and bacterial oxidation processes. The starter embankment for TSF#4 was constructed in 2015 and has the capacity to hold two years' worth of flotation/neutralization tailings. Fosterville currently has at least 5 years of permitted (regulator approved) storage capacity. Therefore Fosterville has a permitted flotation/neutralization storage plan until 2020.

Cyanide bearing leach residue: The leaching circuit uses cyanide to extract the gold and subsequently the liquor possesses traces of cyanide species. As a consequence, the leach residue is deliberately stored separately to that of the flotation residue in a HDPE or clay lined storage facility and only utilized back within the leaching circuits. Tailings is excavated annually from one of the CIL TSF' s and placed onto one of the CIL hardstands. This makes the excavated CIL TSF available for further refilling.

Fosterville has at least 2 years of storage capacity available on existing CIL Hardstands. In 2016, Fosterville will seek regulatory approval for further CIL Hardstand upgrades.

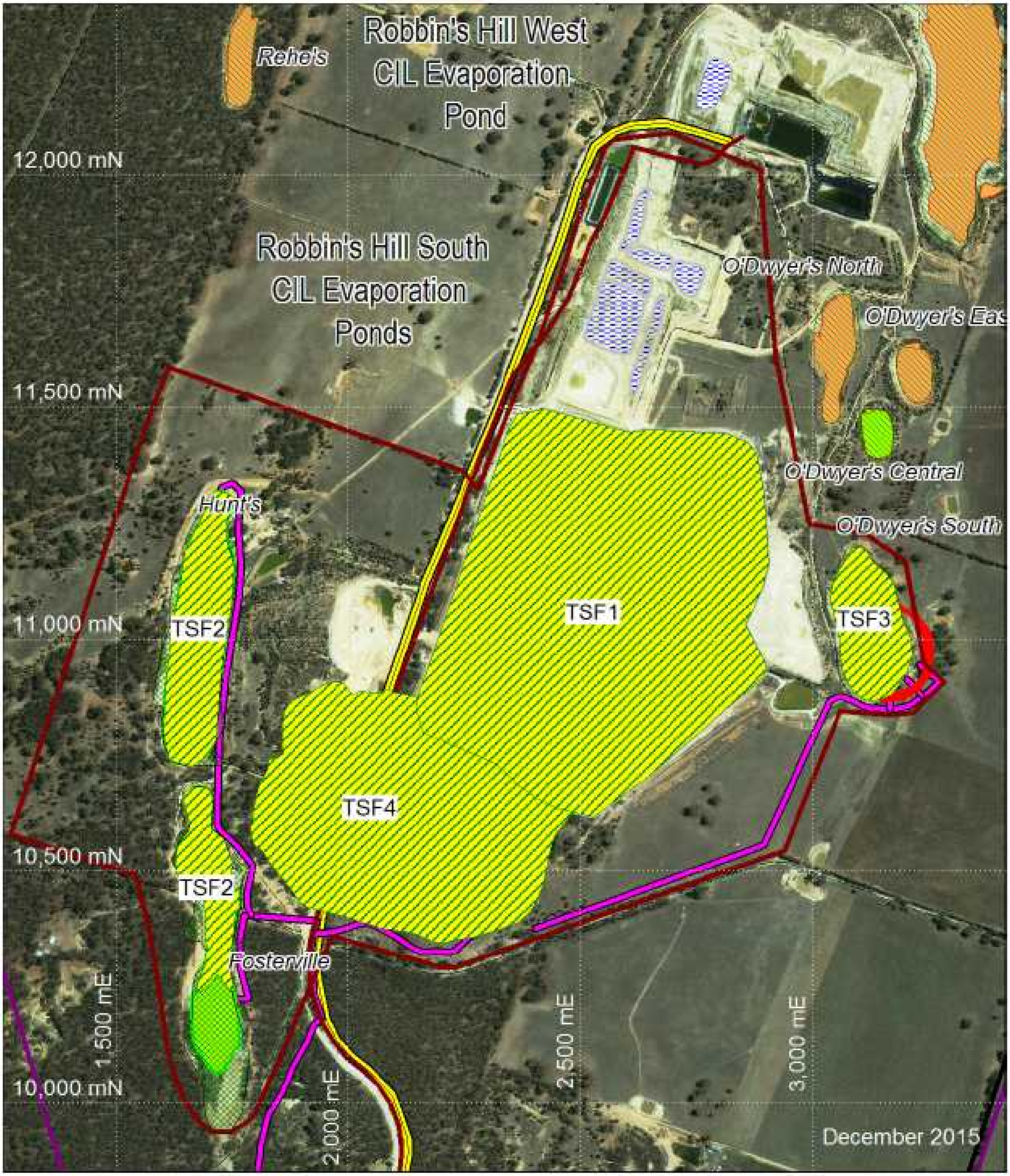

Figure 18-3 Fosterville Flotation and Neutralization Residue Storage Area Plan.

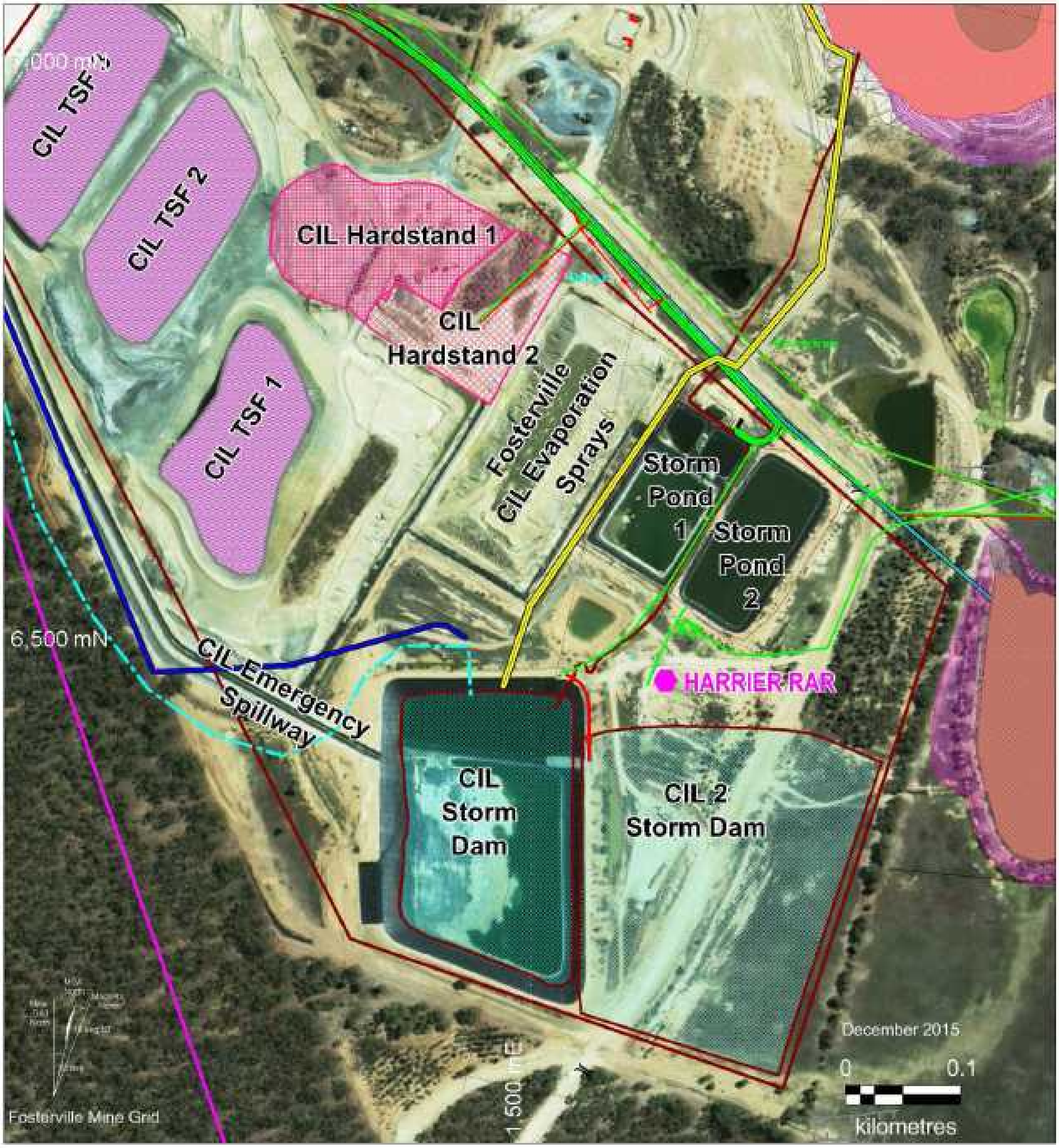

Figure 18-4 Fosterville CIL Residue Storage Area Plan.

18.2 Underground Infrastructure

18.2.1 Power

Power for the underground operations is drawn from Pole 9 11kV Switch Room that connects to the Fosterville Terminal Station (FVTS) Transformer located adjacent to Daley' s Hill.

Three 11,000 volt feeds each enter the Underground workings at

Harrier at the 4775 Sub Station, via the Harrier Vent shaft
Phoenix at the 5031 Sub Station, via a service hole
Ellesmere at the 4968 Sub Station, via a service hole

From these locations low voltage (1000 volt) is reticulated to the working areas via cable and distribution boxes. Further 11,000 volt sub stations are cascaded from the above named primary points as mine working load requires.

Existing underground power reticulation has been sized to meet the designed LOM requirements.

18.2.2 Water

Dewatering of the Fosterville underground workings is conducted utilizing two pumping stations.

Each of these stations comprises of three by WT088 helical rotor pumps that are fed from purpose constructed feed dams.

The Phoenix/Central area is serviced by a station situated at the 4830 level, this station pumps directly to the surface via a steel rising main line that is run through service holes and mine workings and discharges into the Falcon Pit caving area for final settlement of mine solids so that the water can then be utilized within the mine water reticulation system.

The Harrier area is serviced by a station situated at the 4775 level, this station pumps directly to the surface via steel rising main that is run through service holes, mine workings and the Harrier vent shaft and discharges to the Harrier pit.

Mine water is managed through sumps that are, where possible, connected by drain holes, otherwise pumps are used to move water to collection points where it enters staged pumps that transport water from the working areas of the mine to the pump station feed dams. Pumps used for the staged transfer of water are of the helical rotor type, predominantly WT103 type.

Underground mine process water is recycled from the mine water and is reticulated to the underground working areas from a tank farm on the surface.

18.2.3 Ventilation

Primary ventilation of the Fosterville underground workings is achieved utilizing three return air systems, fresh air is drawn into the mine workings via the Falcon and Ellesmere portals.

Central/Phoenix

- Utilize a shared system that exhausts through the Harrier ventilation shaft.
  - 1 x Howden 1500/2400 axial fan situated within the Harrier workings draws air through a series of rises and horizontal development that at present terminate at the Phoenix 4245 level.

Phoenix

- 1 x FlaktWoods TR-1400-GV-4P fan situated underground at the Phoenix 5071 level draws air through a series of rises and horizontal development to maintain flow through the underground magazines. Exhaust is via a rise to the Falcon pit.

Harrier

- Up to 4 x FlaktWoods TR-1400-GV-4P fans are situated underground and draw air through a series of rises and horizontal development that at present terminate at the 4350 level. Exhaust to the surface is via the Harrier ventilation shaft.

Secondary ventilation is provided to the mine working areas utilizing electric fans and flexible ducting. Fans are sized according to air flow requirements and range in size from 22 to 180 kW.

18.2.4 Dumps

Waste material that cannot be placed underground is brought to the surface and dumped within the confines of the Ellesmere pit.

19 MARKET STUDIES AND CONTRACTS

19.1 Markets

Fosterville produces gold doré bars mine site, which are transported to The Perth Mint in Western Australia and refined to produce gold bullion. The gold bullion is sold over the counter according to the corporate treasury policy through either The Perth Mint or an Australian based bank.

To determine the Australian denominated gold price to use in the Mineral Resource and Mineral Reserve calculations, reference was made to publicly available price forecasts by industry analysts for both the gold price in US dollar terms and the AUD/USD foreign exchange rate.

This exercise was completed in November 2015, and yielded the following average gold forecast prices and corresponding average forecast US$:AUD$ FX rates.

For Mineral Reserve purposes, a US$1,100/oz gold price was used and an FX rate of $0.76 for an approximate Australian dollar gold price of AUD$1,450 per ounce.

The average US$ gold price per ounce for the last three years was as follows:

2013	-	US$1,411
2014	-	US$1,266
2015	-	US$1,146

19.2 Contracts

Fosterville is subject to a licence fee following a licence agreement entered into with Biomin South Africa Pty Limited (Biomin) (formally known as Minsaco) in 2003. Biomin has a licence from the proprietor to implement a process known as the BIOX® process in Australia whereby micro-organisms are used in the oxidation of certain gold bearing sulphidic minerals in order to facilitate gold recovery. Fosterville agreed to pay a licence fee to Biomin calculated as an amount determined by multiplying the number of ounces of gold produced from FGM treated through the BIOX® Plant by $1.33. The licence fee was payable from the date of commencement of operations and shall terminate when 1,500,000 ounces of gold in the aggregate has been produced from FGM treated at the BIOX® plant.

When Crocodile Gold acquired the Fosterville and Stawell Gold Mines from AuRico in 2012, a net free cash flow sharing arrangement was established where Crocodile Gold was entitled to cumulative net free cash flow from those mines of up to C$60 million. AuRico would then be entitled to 100% of the next C$30 million in net free cash flow, after which Crocodile Gold and AuRico would share the next C$30 million of net free cash flow on a 50/50 basis until C$120 million of cumulative net free cash flow was achieved, following which AuRico would be entitled to 20% on an ongoing basis.

On December 22, 2014 it was announced that Crocodile Gold had reached a mutually beneficial agreement with AuRico that terminated their net free cash flow sharing arrangement in exchange for a one-time payment of $C20 million in cash and a net smelter return royalty of 2% from Fosterville (effective upon final approval from the Foreign Investment Review Board of Australia) and a 1% royalty from the Stawell Gold Mines (commencing January 1st 2016), releasing Crocodile Gold from its obligation to pay AuRico any further net free cash flow generated from its Victorian operations. This agreement means that Newmarket Gold is obligated to pay AuRico a net smelter royalty of 2%. However, Alamos Gold Inc. (Alamos) merged with AuRico Gold in July 2015, which has resulted in Newmark Gold now being obliged to pay w company, AuRico Metals, the net smelter royalty of 2% from Fosterville Gold Mine.

Fosterville is an owner / operator business with mining, processing, technical and administration functions undertaken by personnel employed by Newmarket Gold. Supplementary support to the operation is sourced through various service contracts. The most significant service contracts include:

E.B. Mawson & Sons Pty. Ltd. – providing services and supply of concrete products.

Fullboar Mining & Maintenance Pty. Ltd. – providing underground drilling services.

Hoare Bros. Pty. Ltd – providing surface haulage services.

Deepcore Australia Pty. Ltd. – providing underground diamond drilling services.

The terms and rates of these contracts are within industry norms. The Authors are not aware of any other agreements that are not within normal market parameters.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Studies and Related Issues

Environmental studies conducted at FGM related to environmental issues are outlined below:

Environmental Noise Assessment - AECOM Acoustic Camera Results (December 2014)

AECOM consultants were commissioned to conduct an acoustic assessment of Fosterville, including measurement and modeling of the operations and provided advice for potential noise mitigation. One of the recommendations was to conduct noise imagery investigations on complicated noise sources within the processing plant using an acoustic camera. The acoustic camera captures a photographic image overlaid by colored sound intensity contours emitted from objects.

The imagery from the Acoustic camera identified the main noise emission areas of discreet plant items. Investigations indicated that the cooling equipment inside the SAG mill lube shed produced a dominant noise source. Changes were made to the cooling fan arrangements within the lube shed and a sound attenuation box was fitted to the lube pump that resides in the shed. Further noise control measures for the Blower building is planned for 2016.

Underground Aquifer Injection

Fosterville Gold Mine produces an excess of mine water from the dewatering of underground operations. Regulatory approval has been gained to treat excess mine water using a Reverse Osmosis plant for reuse in the processing plant. Further studies are being conducted to investigate the potential storage of the treated water via underground aquifer injection. A Work plan Variation for this project is presently being compiled and will be submitted during 2016.

Anaerobic Wetland Pond Trial

Fosterville Gold Mine constructed a small trial anaerobic wetland to treat mine water. The wetland uses sheep manure that contains anaerobic bacteria that are able to treat mine water by biological sulphate reduction. If the project is successful, FGM may construct a larger scale constructed wetland to remediate mine water for potential re-use or underground injection.

CIL and Mine Water Evaporation Spray Monitoring Programs

Environmental monitoring is conducted at the Robbins Hill and Fosterville CIL evaporation facilities as per the CIL management plan. Monitoring is also conducted at the Falcon Pit mine water evaporation facility in accordance with the approved Work Plan Variation. Monitoring includes vegetation assessments, soil monitoring and spray drift catcher monitoring to determine if the operation of the sprays is having any impact on the environment.

CIL Overspray Incident Follow-Up Monitoring

ENSPEC Pty Ltd. was engaged to conduct follow up inspections of trees that had been identified in 2013 to have been damaged by the CIL overspray incident. The inspection identified that most trees were visually observed to display good or low canopy vigor. The improved canopy health of these trees has increased their estimated life expectancies and reduced their additional inspection requirements.

Follow up soil sampling was also undertaken by Sharpe and Howells and those results were reviewed by Golder Associates. The review compared recent and historical data (report undertaken by SKM) to determine the current risk to the environment and animal health. The review found the land was suitable for its previous land use, horse grazing.

Biosolids Trial

In collaboration with Coliban Water Fosterville Gold Mine is conducting a biosolid fertilizer trial. Biosolids is a solid product from sewage treatment processes and have been treated in a way to make them safe for further use. The biosolid fertilizer has been incorporated into a number of soil plots and planted with native species. Monitoring of the trial plots over time will determine the effectiveness of the biosolid fertilizer in improving soil structure and fertility and also enhancing vegetation establishment and growth.

Monitoring of the plots is being undertaken by Goldfields Revegetation Ltd. Early stages of monitoring indicate the best plant growth and visual appearance of vegetation was observed in the plot containing a mix of waste rock and biosolids. However, overall the impact of applying biosolids on the native vegetation appears to be minimal.

Heap leach Environmental Characterization

An environmental characterization of Robbins Hill heap leach waste was undertaken to assist determination of the appropriate closure strategy for the material. The potential re-use of heap leach as a capping material has potential environmental and cost benefits by reducing rehabilitation liability and providing a readily available source of construction material.

Sampling was carried out during August 2015 from 5 cores drilled in heap leach waste rock. The purpose of the sampling regime was to identify changes in the geochemical properties of the heap leach with depth. Data analysis is being undertaken and a draft report is being prepared in Q1 2016.

AECOM Dust Dispersion Study (2015)

A Dust Dispersion Study was undertaken to review the effectiveness of the existing monitoring locations using dispersion modeling. Recommendations were made to improve the existing monitoring network and are being considered for 2016.

20.2 Waste and Tailings Disposal, Site Monitoring and Water Management

20.2.1 Requirements

Requirements for residue storage sites are provided in the following documents:

- Section 4.5 of the 2004 Work Plan
- Approved Work Plan Variation for *Additional Portal Access Points (three in total), additional CIL storage facilities (including on the Fosterville Heap Leach Pad) and the construction of a reload facility* (February 22, 2005)

Work Plan Variation CIL Tails Storage and Decant Water Management (July 1, 2008)
Work Plan Variation CIL Residue Hardstand Area (October 23, 2009)
Work Plan Variation, “In-Pit Residue Disposal Facility” (November 2009)
Work Plan Variation, CIL Residue Hardstand #2 Area (March 2012)
Work Plan Variation, “In-Pit Residue Disposal Facility – TSF3 O’ Dwyer’ s South Pit” (November 2012)
Work Plan Variation, “Raising of existing embankment of TSF1” (December 2013)
Work Plan Variation, “Additional Residue Storage Facility - TSF4” (September 2014)

Flotation and Neutralization Tails

Flotation and neutralization tails have been stored in the following facilities:

TSF1,
Hunts and Fosterville In-Pit Facilities, and
O’ Dwyer’ s South In-Pit Facility.
TSF4

During 2015 FGM deposited flotation and neutralization tails into TSF1, Hunts in pit facility, O’ Dwyer’ s South In-pit facility and a new above ground tailings storage facility that was commissioned in November 2015 (TSF4). .

The Fosterville In-Pit Facility has been filled and capped. Capping performance is being monitored by the amount of rainfall infiltration through the cap, and is measured by two lysimeters installed within the cover profile. The Hunts In-Pit facility has approximately six months of tailings storage remaining.

CIL Tailings

All CIL tailings have been stored in lined tailing facilities within and adjacent to the old Fosterville Heap leach facility.

The combination of surface impoundment dams and hardstand areas provide sufficient capacity to receive CIL tailings for the remainder of the current LOM

Overburden Waste

The deposition/distribution of overburden waste throughout the Fosterville site is outlined in Table 20-1

Table 20-1 Overburden use at Fosterville Gold Mine

Overburden Source	Use
Falcon Pit	Construction of TSF1 (internal rock armoring of walls) Construction of the ROM pad Construction of haul roads Backfill into Vanessa' s North Pit and at the southern end of Fosterville Pit (the remainder is flotation tailings) Construction of McCormicks Waste Dump (majority)
Ellesmere Pit	Sound bunds on the eastern side of Ellesmere (possibly Harrier sound bund as well) McCormicks Waste Dump Falcon Backfill
Johns Pit	Backfilling Harrington Hill South Open Pit[1] Backfilling into Harrier Open Pit (western side) Use for repairing the CIL Storm Dam wall Abandonment bund walls for Johns Pit South end of Ellesmere
O'Dwyer's South Open Pit	To be used as backfill into the northern end of the Pit To be placed into the existing O'Dwyer's South Waste Dump
Harrier	Backfilling into Ellesmere Pit south to north Construction of internal ramps in Harrier Pit Sound walls to the east of Harrier Pit To be used for rock fill for CIL #3
Hunts	TSF1[2] main embankment Building Hunts Pit Waste Dump
Fosterville	Hunts Waste Dump
Underground	Backfilled into underground workings Used as base in the Ellesmere Saddle

Notes:

1. Sediment from Fosterville Storm Dam was also transferred into Harrington Hill South Pit.

2. TSF1 was also constructed using heap leach material from Robbins Hill.

Potentially Acid Forming Materials

Known potentially acid forming (PAF) materials excavated from open pits have been stored in:

McCormick' s Waste Dump;
Johns Pit (taken from Johns Pit and Harrier Pit); and
Flotation and Neutralization Tailings.

FGM undertook a gap analysis and operational phase waste rock characterization during 2013 and 2014 to validate AMD predictions in new mining areas, and assist mine closure planning. The results showed that waste rock was overall non-acid-forming and contained a significant inherent Acid Neutralizing Capacity that is available to offset any isolated acid formation. Kinetic column leach testing of the main waste rock lithologies is presently being undertaken. A Waste Rock Management Plan was developed. It is intended that further waste rock samples will be taken during 2016.

20.2.2 Site Monitoring and Water Management

Water Management

The Fosterville annual water monitoring plan is designed to monitor the impacts of mining activities on surface and groundwater quality and quantity in the regional and local aquifer systems. Water samples are collected on monthly, quarterly or an annual basis in accordance with the Work Plan (2004) and the annual water monitoring schedule which is reviewed each year.

Groundwater levels in the monitoring bores are also recorded each month.

Noise Monitoring

Noise monitoring is undertaken in accordance with the Work Plan (2004) and Work Plan variation (2015) and includes periodic day, evening and night measurements at nine representative locations surrounding the mine. Noise results are assessed against EPA criteria and any mine related exceedances are reported to the Regulators.

Air Quality

Dust deposition rates were monitored on a monthly basis at 10 sensitive receptors around the mine. The quantity of material deposited was analyzed for total insoluble material (g/m^2), which comprises non-combustible material (ash) and combustible material. Ash content provides an indication of the mineral content of a sample. The mineral content may be attributable to mining, but may also be attributable to other sources such as agriculture, unsealed roads etc. The combustible material will not be attributable to mining as this is mostly organic matter.

Dust was also measured at a sensitive receptor by a high volume sampler. The high volume sampler measures the particulate loading in the air less than 10 and 2.5 microns (mg/m^3).

Greenhouse gases and other emissions are evaluated and reported under the National Greenhouse and Energy Reporting and National Pollutant Inventory regulatory programs on an annual basis.

Rehabilitation monitoring

As part of the Environmental Management Plan, Fosterville undertakes progressive rehabilitation of areas affected by the operations, taking into consideration the future end use of the land. Progressive rehabilitation includes stabilization earthworks, drainage enhancement and control works, establishing vegetation, weed and pest animal control and continual monitoring. Bi-annual monitoring of the revegetation works associated with the McCormick’ s Waste Dump site and the O’ Dwyer’ s South Pit remnant patch is conducted by an independent consultant. The scope of the monitoring includes an assessment on plant growth and survival, threats to plant survival and the presence of pest plants and animals.

Vibration Monitoring

Blast monitoring was undertaken at a sensitive receptor outside the boundary of the Fosterville Mine Lease with a permanently installed blast monitor. All of the blasts that were monitored during 2015 were within the mining licence limits.

20.3 Project Permitting Requirements

Fosterville currently operates under a Mining Lease and Mining Licence dated 2003. A Work Plan was approved for the project in 2nd of February 2004. There have been a number of Work Plan Variations that have been prepared for the project which form addendums to the 2004 Work Plan. Recent Work Plan Variations approved include:

- Mine water evaporation sprays in North end of Falcon Pit (August 2014)
- Mine water treatment plant (September 2014)
- Additional Residue Storage Facility - TSF4 (September 2014)
- Reviewed Noise Monitoring Limits (February 2015)

A Work Plan Variation for underground aquifer injection (see Section 20.1) is presently being compiled and will be submitted during 2016.

There are a number of requirements relating to rehabilitation and closure both in the Licence, the 2004 Work Plan and in subsequent Work Plan Variations. All rehabilitation and closure requirements have been incorporated into the site' s Closure Plan.

20.4 Social or Community Related Requirements and Plans

Community engagement and consultation on all aspects of the operation continues as an integral part of the FGM business model. There are a range of forums and consultation undertaken - from quarterly Environmental Review Committee Meetings and the annual Public Meeting to a range of project or activity-specific meetings where future activities and plans are communicated. The feedback from these sessions is utilized in planning any future projects. Fosterville Gold Mine also has a Community Engagement Plan and prepares an annual Environment and Community Report that is made available to all members of the community.

One project that has involved a long and extensive consultation with the local community has been the replacement of the original Fosterville Huntly Road, which was altered to allow for the development of the Falcon pit. Fosterville, in collaboration with the City of Greater Bendigo negotiated a road realignment which was acceptable to the community, in lieu of the original alignment and the construction was successfully completed in December 2014. The road was officially opened on 23rd January 2015.

A mine closure information presentation evening was provided for the community in June 2015, which provided information on various projects relating to land use functionality, final land end use, monitoring and rehabilitation requirements into the future.

20.5 Mine Closure (Remediation and Reclamation) Requirements and Costs

The most recent bond lodged with the Department of Economic Development, Jobs, Transport and Resources (DEDJTR) is AUD$7.835M. Acceptance of this bond is expected in early 2016. All closure requirements are included in the FGM Closure Plan.

Key operational domains for reclamation works include:

- Northern Site Facilities,
- Southern Site Facilities,
- Sulphide Infrastructure,
- Sulphide Open Pits,
- Adits and Shafts,
- Main Overburden Heap,
- Tailings Storage Facility,
- CIL Dams,
- Heap Leach Pads,
- Oxide Open Pits.

After an investigation into the potential realization estimates of the FGM assets, including the processing plant, ancillary equipment, non-fixed assets and the mining mobile fleet, the Company considers the current processing plant as a valuable asset that will be able to be successfully sold as an entire operation unit and removed down to the foundations on a cash positive basis. The demolition of the plant is therefore an integral cost within the Rehabilitation Bond Provision at this time.

In addition to disposal of the plant, key closure activities for FGM include:

- Decommissioning and rehabilitation of the heap leach facilities, associated dams and infrastructure.
- Decommissioning and rehabilitation of the tailings facilities (including TSF1 and the in-pit storages).
- Decommissioning and rehabilitation of the CIL tails facilities and associated dams.
- Rehabilitation of old open pits, and
- Revegetation of all remaining disturbed areas.

21 CAPITAL AND OPERATING COSTS

21.1 Capital and Operating Estimates

The capital and operating costs for the FGM are presented below in Table 21-1.

The basis of the below estimates is on operating history and known increases in cost for the current and future years.

Operating Costs

All 2016 costs as per budget

2017/18/19 operating costs as per 2016 reflecting a similar production profile and operating development schedule. This assumes that ongoing cost control/productivity program can keep pace with inflation.

Operating costs for the reduced volume in 2020 have been estimated at $70/t for mining (mostly stoping only), $3/t for geology, $45/t processing and $14/t for administration. Residual administration costs during tails-only operation have been estimated at $100/oz post mine closure.

Tails retreat-only processing costs (post-mining) have been estimated at $900/oz (+100/oz for administration).

Royalty is reflected as 2% of revenue throughout the model.

Capital Costs

All 2016 costs as per budget

Mobile Plant and Equipment has been modeled as $5.9m annually for 2017- 19 reflecting the annual average from 2012-16.

Processing Plant capital modeled at 2016 budget levels for the period 2017-19.

Infrastructure capital has been increased from 2016 levels by $3.3m in 2017 ($2m mine water, $1.3m for TSF1 Lift 6) and by $3.2m in 2018 (TSF4 Lift 2).

Underground Development capital modeled at 2016 budget levels 2017- 19 reflecting the development required (decline, level accesses, ventilation raises) to access the subsequent year of production.

Resource Definition capital in 2016 has been increased from recent levels in order to target 150% replacement of reserve ounces mined in 2016. Drilling for the period 2017- 19 has been reduced to $10m per annum reflecting recent replacement cost of reserves mined (~$80/oz).

Table 21-1 Capital and operating cost estimates from the Current LOM plan.

Fosterville Gold Mine Current LOM Reserves + Resource Conversion	LOM	FY-16	FY-17	FY-18	FY-19	FY-20	FY-21
Operating Costs							
Surface Mining	$0	$0	$0	$0	$0	$0	$0
Underground Mining (includes geo & mine maint.)	$217,954,316	$45,363,579	$45,363,579	$45,363,579	$45,363,579	$36,500,000	$0
Processing (includes refining, transp. & mill maint.)	$156,244,228	$26,461,057	$26,461,057	$26,461,057	$26,461,057	$22,500,000	$27,900,000
Administration	$47,012,036	$9,228,009	$9,228,009	$9,228,009	$9,228,009	$7,000,000	$3,100,000
Royalty	$16,067,446	$3,237,638	$3,163,186	$3,163,186	$3,163,186	$2,441,251	$899,000
	$437,278,026	**$84,290,283**	**$84,215,831**	**$84,215,831**	**$84,215,831**	**$68,441,251**	**$31,899,000**
Capital							
Mobile Plant and Equipment	$23,470,869	$5,770,869	$5,900,000	$5,900,000	$5,900,000	$0	$0
Mobile Plant and Equipment Under Finance	$2,940,000	$2,940,000	$0	$0	$0	$0	$0
Processing Plant	$2,700,000	$675,000	$675,000	$675,000	$675,000	$0	$0
Infrastructure	$11,140,000	$1,160,000	$4,460,000	$4,460,000	$1,160,000	$0	$0
Land and Buildings	$0	$0	$0	$0	$0	$0	$0
Underground Development	$114,475,036	$28,618,759	$28,618,759	$28,618,759	$28,618,759	$0	$0
Surface Development	$0	$0	$0	$0	$0	$0	$0
Resource Definition	$45,235,090	$15,235,090	$10,000,000	$10,000,000	$10,000,000	$0	$0
Exploration	$0	$0	$0	$0	$0	$0	$0
	$199,960,995	**$54,399,718**	**$49,653,759**	**$49,653,759**	**$46,353,759**	**$0**	**$0**

22 ECONOMIC ANALYSIS

As per Item 22: Economic Analysis, Instruction 1, item 22 has been excluded on the basis that the property is currently in production and there are no plans for material expansion of current production.

23 ADJACENT PROPERTIES

As shown in Figure 4-2, the Fosterville Mine Lease (MIN5404) is completely enveloped by exploration leases held by Newmarket Gold (through Fosterville Gold Mine Pty Ltd). There is Fosterville-style-mineralization identified in the Goornong area (5km to the north of MIN5404) and the Hallanan's area (2km to the south), as discussed in van Riel (1999) However, the exploration of these prospects is only at an early stage and not relevant to discuss further in relation to this Technical Report.

No other Fosterville-style gold operations are in production in the Fosterville district. However, Fosterville-style mineralization does occur in the Lockington area (Boucher et al, 2008b and Arne et al, 2009), 50 km north of Fosterville where eight mineralized trends have been mapped beneath thick cover using aircore drilling. The project is at an early exploration stage.

24 OTHER RELEVANT DATA AND INFORMATION

No other relevant information is required to make the technical report understandable and not misleading.

25 INTERPRETATION AND CONCLUSIONS

The authors have made the following interpretations and conclusions:

The understanding of the fundamental geological controls on mineralization at Fosterville is high. Primary mineralization is structurally controlled with high-grade zones localized by the geometric relationship between bedding and west dipping faulting. This predictive model has led to considerable exploration success in following the down-plunge extensions of high-grade mineralization.

- The **Lower Phoenix Fault** is the primary west dipping structure in the active mine development area and is defined by reverse faulting on a shale package where anticline thrust displacement of ~80 meters occurs. The fault dips between 35 and ~55 degrees to the west and mineralization can be traced along a dip extent of ~190m and strike extent of ~1.3km. The dominate mineralization style on this structure is disseminated sulphide, however occurrences of visible gold at depth are becoming increasingly common, concentrated where footwall structures intersect. The Lower Phoenix System currently remains open to the north and south so maximum plunge extent has not yet been defined.

- Throughout 2015, development mapping and continued drilling confirmed that there were multiple mineralized structures of various size and continuity footwall to the main west dipping Lower Phoenix Fault, which present significant resource growth potential. Progressive geological understanding of the Phoenix and Lower Phoenix footwall environs has highlighted the significance of these favorable settings for mineralization, including:

  East Dipping mineralized structures, namely the **Eagle Fault** and **East Dipping Faults** which commonly contain quartz- stibnite vein assemblages and substantial concentrations of visible gold typically enveloped by halos of disseminated sulphide. The Eagle Fault is discordant to bedding and variably dips between 10 and 60 degrees to the east, where East Dipping Faults are typically bedding parallel to sub parallel with dips of ~70 degrees east to sub vertical.

  Low- angled **Lower Phoenix Footwall** west dipping structures typically consist of large quartz veins up to several meters wide with laminated textures, indicating a series of multiple mineralizing events, including a later stage quartz-stibnite phase of mineralization and visible gold. The faults are interpreted to have minimal offset but rather have been hydraulically fractured. Where these structures form linkages between the Lower Phoenix and East Dipping Faults, extremely high gold grades are observed.

  Continued drill definition of these structures over 2015, in combination with ore development and production exposure and reconciliation performance has reaffirmed the significance of these easterly dipping footwall structures to the Lower Phoenix Fault. The defined continuity, proximity to existing Mineral Resources and high grade tenor of these structures enhances the December 2015 Mineral Resource and Reserve position. Furthermore, mineralization on these structures is open down plunge, providing encouraging future Mineral Resource and Mineral Reserve growth potential for the operation.

There is an observed change in the nature of some of the Fosterville mineralization at depth, with a number of high- grade, quartz-stibnite hosted, visible gold drill intercepts recorded for the Eagle, Lower Phoenix, Lower Phoenix Footwall and East Dipping Zones. Disseminated sulphide mineralization continues to persist at all depths and is uniform in character. It is currently inferred that the quartz-stibnite- visible gold assemblages have been emplaced at a later date to the disseminated sulphide providing an upgrade to the mineralization.

In addition, a better understanding of the mineralization of the **Kestrel System** was established during 2015. Drilling has defined an extensive broad zone of low to moderate grade disseminated sulphide mineralization (~average 6m in width) centered around the syncline hinge axial plane.

Progressive geological interpretation has led to continued development of robust geological and resource models underpinning the Mineral Resource and Mineral Reserve estimates. The relationship between mineralization and the controlling structural/stratigraphic architecture means that quality geological interpretation is critical to producing quality resource/reserve estimates.

The modifying factors used to convert the Mineral Resources to Mineral Reserves have been refined with the operating experience gained since underground production commenced in September 2006. In particular, the robustness of the mining recovery and dilution estimates has improved with experience relative to the pre- mining assessments.

Fosterville Gold Mine has a demonstrated solid production history over a 10 year plus period since the beginning of commercial sulphide gold production in April 2005, and it is the authors view that the risk of not achieving projected economic outcomes is low given the operational experience gained over this time period. A foreseeable risk and uncertainty facing the operation is the changing character of mineralization at depth with an increase in the occurrence of visible gold. Reconciliation results in the past have provided confidence in the sample collection procedures, the quality of assays and the resource estimation methodology, but these processes will need to be continually adapted in consideration of the changing mineralization character at depth. Newmarket Gold needs to continue research to better understand the potential implications on future geological, mining and metallurgical processes and will continue to seek external advice over 2016 in relation to sampling, assaying and mineral resource estimation of visible gold mineralization. Based on recommendations from previous external reviews, projects plans have been developed and implemented.

26 RECOMMENDATIONS

The following recommendations are made:

Further Fosterville Mine Lease growth exploration activities should be pursued. Given the strong understanding of geological controls on mineralization, this could have the potential to yield additional resources and reserves. Particular areas that are recommended to focus upon are the up and down-plunge extensions of the Lower Phoenix structure (7750mN and northwards and 6200mN and southwards). Further 'step out' on lease exploration targets should also be pursued and form part of the 2016 exploration budget, including the 5450mN Lower Phoenix targets (Figure 26-1). The current 2016 exploration budget (Table 26-1) includes a combined total of 14,000m of diamond drilling for an estimated AUD$3.35M to explore these gold targets. Approximately 320m of hanging wall drill drive development from the Harrier Decline at an estimated cost of AUD$2.95M forms part of the 2016 budget plan to facilitate the down plunge exploration of the Lower Phoenix System south of 6200mN.

Other attractive on lease targets to be explored in future planning include follow up drilling on the O' Donnell' s Line (approximately 1km to the east of the Fosterville Line of workings) and the northwest portion of the Robbins Hill Pit.

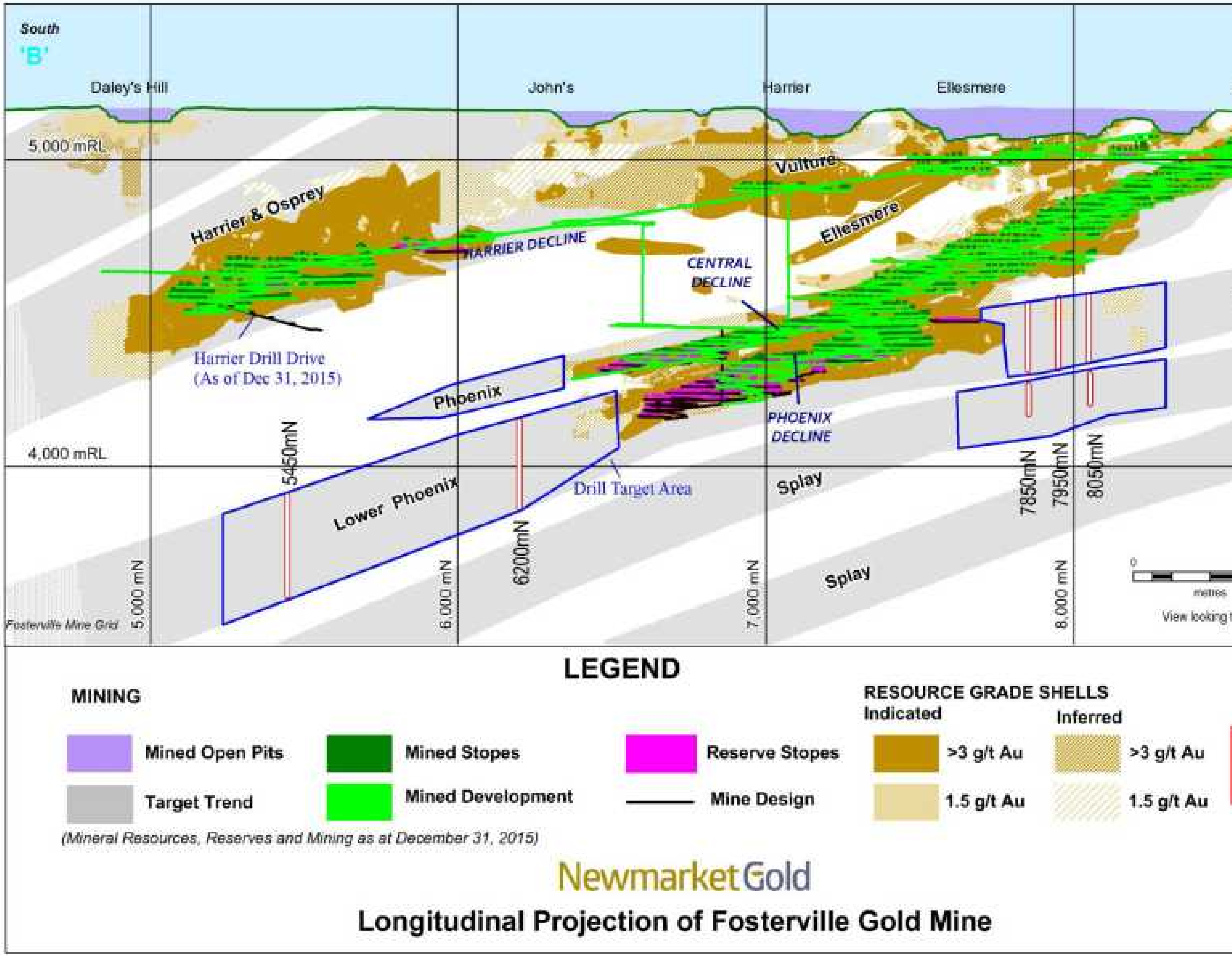

Figure 26-1 Longitudinal Projection of proposed Exploration drilling programs for 2016.

Table 26-1: Proposed drilling programs for 2016.

Exploration programs for 2016	2015 Exploration status	Forecast End of 2016 Exploration status	Description	Program details		Expenditure (AUD)
				Total number of holes	Total metres	
Lower Phoenix South (5450mN)	Geological Target	Investigative	Targets the projected down-plunge extension of the currently known Lower Phoenix mineralization	1	2,240	$ 460,000
Phoenix-Kestrel South (5450mN)	Geological Target	Investigative	Targets the projected down-plunge extension of the currently known Phoenix & Kestrel mineralization	7	3,350	$ 600,000
Anticline-Syncline South (5450mN)	Geological Target	Investigative	Targets the projected location of the hinges associated with the Lower Phoenix, Phoenix, & Kestrel mineralization	5	1,020	$ 240,000
Lower Phoenix North (7850mN)	Geological Target	Investigative	Targets the up-plunge extension of the Lower Phoenix mineralization on the 7850mN	4	3,030	$ 715,000
Fosterville Splays North (7850mN)	Geological Target	Investigative	Targets deeper splay faulting related to hinge offset below known mineralized trends on the 7850mN	1	646	$ 260,000
Lower Phoenix North (8050mN)	Geological Target	Investigative	Targets the up-plunge extension of the Lower Phoenix mineralization on the 8050mN	4	2,393	$ 715,000
Fosterville Splays North (8050mN)	Geological Target	Investigative	Targets deeper splay faulting related to hinge offset below known mineralized trends on the 8050mN	1	886	$ 260,000
Benu South (6200mN)	Investigative	Project Scoping	Targets the down-plunge extension of the Lower Phoenix mineralization on the 6200mN. Completion of 2015 Program.	1	435	$ 100,000
Harrier 4180 Drill Drive (5450mN)	NA	NA	Exploration capital development drive to be used for the 5450mN drill programs	NA	320	$ 2,950,000

Prospective regional exploration targets also exist in close proximity to the existing mine lease and form part of the overall future drill planning. These prospects include Sugarloaf, Goornong and Hallanan' s. Other prospects further away from the Mine Lease will not be overlooked and will be explored in future years.

The infill/resource definition programs should be continued with an aim to maintain at least 12 months of reserves drilled out to 25m centers (or closer where necessary). Both the south plunging, westerly dipping Phoenix and Lower Phoenix Mineralized Zones and the easterly dipping Eagle and East Dipping Mineralized Zones require definition drilling which is to be conducted from both hangingwall (western side) and footwall (eastern side) drill platforms. The current infill drilling budget for 2016 includes 64,416m of drilling at an estimated cost of AUD$10.43M. As the decline and mining front continues to move south and lower, further hangingwall drives will be required to be developed. This work and the associated drilling have not been costed in detail.

It is also recommended that infill / resource definition programs target down plunge extensions of the Harrier Mineralized Zones with the aim to increase Mineral Reserves. Aspirational reserve work on Inferred Mineral Resources in the Harrier South area indicated that with smaller scale mining parameters applied these Mineral Resources have the potential to be converted into Mineral Reserves that would have high potential to increase the current LOM at FGM. Newmarket has incorporated into its 2016 budget planned drilling into the Harrier South area with the aim to increase mineral resource confidence to allow for mineral reserve evaluation. This budgeted phased drill program consists of 32,208m at an estimated cost of AUD$5.21M.

With this additional drilling data and further ongoing operational experience, it is recommended that mining recovery and dilution factors are reviewed and refined on an ongoing basis.

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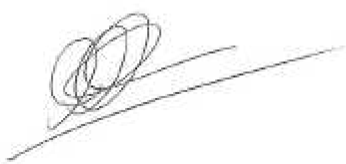
28 DATE AND SIGNATURE

28.1 Certificate of Qualified Person – Ion Hann

As a co-author of the report titled: “Report on the Mineral Resources & Mineral Reserves of the Fosterville Gold Mine Victoria, Australia”, prepared for Newmarket Gold Inc. (“Newmarket Gold”), effective December 31, 2015 and dated March 21, 2016, (the “Technical Report”) to which this certificate applies, I, Ion Hann, FAusIMM , do hereby certify that:

1. I am a Mining Manager of Fosterville Gold Mine, employed by Newmarket Gold. The address of Fosterville Gold Mine is McCormicks Road, Fosterville, Victoria 3557, Australia.
2. I graduated in 1991 from the Western Australian School of Mines (WASM), a regional campus of Curtin University, with a Bachelor of Engineering (Mining) degree.
3. I am a Fellow in good standing of the Australasian Institute of Mining and Metallurgy (Member No. 302934).
4. I have worked for more than 20 years in underground mining, including more than 10 years in gold mining operations.
5. I work directly for Newmarket Gold in the role of Mining Manager and my work is based at the Fosterville Gold Mine and I am not independent of Newmarket Gold pursuant to National Instrument 43-101 (“NI43-101”).
6. I am responsible for the Items 15-16, 18.1 and 28.1of the Technical Report.
7. I have read the definition of “Qualified Person” set out in NI43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a “Qualified Person” for the purposes of NI 43-101.
8. I have read NI43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 21st day of March, 2016.



Ion Hann, B.Eng (Mining), FAusIMM

MINING MANAGER

FOSTERVILLE GOLD MINE

28.2 Certificate of Qualified Person – Troy Fuller

As a co-author of the report titled: “Report on the Mineral Resources & Mineral Reserves of the Fosterville Gold Mine Victoria, Australia”, prepared for Newmarket Gold Inc. (“Newmarket Gold”), effective December 31, 2015 and dated March 21, 2016, (the “Technical Report”) to which this certificate applies, I, Troy Fuller, MAIG, do hereby certify that:

1. I am the Geology Manager of Fosterville Gold Mine, employed by Newmarket Gold. The address of Fosterville Gold Mine is McCormicks Road, Fosterville, Victoria 3557, Australia.
2. I graduated in 1995 from the University of Ballarat with a Bachelor of Applied Science (Geology) Honours degree.
3. I am a member in good standing of the Australasian Institute of Geoscientists (Member No. 4570).
4. I have worked for more than 20 years in the mining industry, including more than 17 years in gold mining operations. I am familiar with and have worked on a variety of styles of mineral deposits in Australia, with a particular emphasis on gold mineralization.
5. I work directly for Newmarket Gold in the role of Geology Manager and my work is based at the Fosterville Gold Mine and I am not independent of Newmarket Gold pursuant to National Instrument 43-101 ("NI43-101").
6. I am responsible for the Items 1-14, 17, 18.2 - 27 and 28.2 of the Technical Report.
7. I have read the definition of "Qualified Person" set out in NI 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
8. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 21st day of March, 2016.

Troy Fuller, BSc (Geology) Hons, MAIG

GEOLOGY MANAGER

FOSTERVILLE GOLD MINE

Technical Report
Preliminary Economic Assessment
of the Maud Creek Gold Project,
Northern Territory, Australia
Prepared for:
Newmarket Gold Inc
According to National Instrument 43-101 and Form 43-101 F1
Prepared by:
SRK Consulting (Australia) Pty Ltd
ABN 56 074 271 720
Level 1, 10 Richardson St, West Perth, Western Australia, 6005
SRK Project Number: CGC001
Qualified Persons:
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Danny Kentwell, MSc Mathematics & Planning (Geostatistics), FAusIMM, Principal Consultant

Simon Walsh, BSc (Extractive Metallurgy), MBA Hons, MAusIMM (CP), GAICD, Associate Principal Consultant.
Date of Report: **16 May 2016**
Effective Date: **15 April 2016**

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Date and Signature Page

SRK Project Number: CGC001

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West Perth, Western Australia, 6005, Australia

Newmarket Gold Inc

Maud Creek Gold Project

Northern Territory, Australia

Project Manager: Peter Fairfield

Date of Report: 16 May 2016

Effective Date: 15 April 2016

Signature Qualified Persons:

Peter Fairfield, BEng (Mining), FAusIMM, CP (Mining), Principal Consultant

Danny Kentwell, MSc Mathematics & Planning (Geostatistics), FAusIMM, Principal Consultant

Simon Walsh, BSc (Extractive Metallurgy), MBA Hons, MAusIMM (CP), GAICD, Associate Principal Consultant

Important Notice

This technical report has been prepared as a National Instrument 43-101 Technical Report, as prescribed in Canadian Securities Administrators' National Instrument 43-101, Standards of Disclosure for Mineral Projects (NI 43-101) for Newmarket Gold Inc. (Newmarket Gold). The data, information, estimates, conclusions and recommendations contained herein, as prepared and presented by the Authors, are consistent with:

information available at the time of preparation;

data supplied by outside sources, which has been verified by the authors as applicable; and

the assumptions, conditions and qualifications set forth in this technical report.

CAUTIONARY NOTE WITH RESPECT TO FORWARD LOOKING INFORMATION

This document contains forward-looking information as defined in applicable securities laws. Forward looking information includes, but is not limited to, statements with respect to the potential future production/ mill feed, costs and expenses of the Preliminary Economic Assessment (PEA) and project; the other economic parameters, as set out in this technical report, including; the success and continuation of exploration activities, including drilling; estimates of Mineral Resources; the future price of gold; government regulations and permitting timelines; requirements for additional capital; environmental risks; and general business and economic conditions. Often, but not always, forward-looking information can be identified by the use of words such as plans, expects, is expected, budget, scheduled, estimates, continues, forecasts, projects, predicts, intends, anticipates or believes, or variations of, or the negatives of, such words and phrases, or statements that certain actions, events or results may, could, would, should, might or will be taken, occur or be achieved. Forward-looking information involves known and unknown risks, uncertainties and other factors which may cause the actual results, performance or achievements to be materially different from any of the future results, performance or achievements expressed or implied by the forward-looking information. These risks, uncertainties and other factors include, but are not limited to, the assumptions underlying the production estimates not being realized, decrease of future gold prices, cost of labour, supplies, fuel and equipment rising, the availability of financing on attractive terms, actual results of current exploration, changes in project parameters, exchange rate fluctuations, delays and costs inherent to consulting and accommodating rights of local communities, title risks, regulatory risks and uncertainties with respect to obtaining necessary permits or delays in obtaining same, and other risks involved in the gold production, development and exploration industry, as well as those risk factors discussed in Newmarket Gold' s latest Annual Information Form and its other SEDAR filings from time to time. Forward-looking information is based on a number of assumptions which may prove to be incorrect, including, but not limited to, the availability of financing for Newmarket Gold' s production, development and exploration activities; the timelines for Newmarket Gold' s exploration and development activities on the property; the availability of certain consumables and services; assumptions made in Mineral Resource estimates, including geological interpretation grade, recovery rates, price assumption, and operational costs; and general business and economic conditions. All forward-looking information herein is qualified by this cautionary statement. Accordingly, readers should not place undue reliance on forward-looking information. Newmarket Gold and the authors of this technical report undertake no obligation to update publicly or otherwise revise any forward-looking information whether as a result of new information or future events or otherwise, except as may be required by applicable law.

NON-IFRS MEASURES

This technical report contains certain non-International Financial Reporting Standards measures. Such measures have non standardized meaning under International Financial Reporting Standards and may not be comparable to similar measures used by other issuers.

Table of Qualified Persons and Contributors

Section	Description	Nominated QP	Contributors
	Executive Summary	Peter Fairfield	All
2	Introduction	Peter Fairfield	
3	Reliance on Experts	Peter Fairfield	
4	Property Description and Location	Peter Fairfield	
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Peter Fairfield	
6	History	Danny Kentwell	
7	Geological Setting and Mineralization	Danny Kentwell	
8	Deposit Types	Danny Kentwell	
9	Exploration	Danny Kentwell	
10	Drilling	Danny Kentwell	Kirsty Sheerin
11	Sampling Preparation, Analysis and Security	Danny Kentwell	Kirsty Sheerin
12	Data Verification	Danny Kentwell	Kirsty Sheerin
13	Mineral Processing and Metallurgical Testing	Simon Walsh	
14	Mineral Resource Estimates	Danny Kentwell	
15	Mineral Reserve Estimates	Peter Fairfield	
16	Mining Methods	Peter Fairfield	Anne-Marie Ebbels, Scott McEwing
17	Recovery Methods	Simon Walsh	
18	Project Infrastructure	Simon Walsh	Peter Fairfield
19	Market Studies and Contracts	Peter Fairfield	Simon Walsh
20	Environmental Studies, Permitting and Social, or Community Impact	Peter Fairfield	Lisa Chandler, Ken Redwood
21	Capital and Operating Costs	Peter Fairfield	Simon Walsh
22	Economic Analysis	Peter Fairfield	
23	Adjacent Properties	Peter Fairfield	All
24	Other Relevant Data and Information	Peter Fairfield	All
25	Interpretation and Conclusions	Peter Fairfield	All
26	Recommendations	Peter Fairfield	All
27	References	Peter Fairfield	All

List of Abbreviations

Abbreviation	Meaning
2D	two dimensional
3D	three dimensional
AAS	atomic absorption spectroscopy
ALS	ALS Minerals
AMC	AMC Consultants Pty Ltd
Amdel	Amdel Limited Mineral Services Laboratory
ANFO	ammonium nitrate-fuel oil
ASL	above sea level
ATCF	after tax cash flow
Au	gold
Au	gold equivalent
AUD	Australian dollar
BAppSc	Bachelor of Applied Science
BCom	Bachelor of Commerce
BD	bulk density
BEng	Bachelor of Engineering
BIOX®	Bacterial Oxidation plant BIOX
BSc	Bachelor of Science
Cambrian	Cambrian Mining Limited
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIP	carbon-in-pulp
(CP)	Chartered Professional of The Australasian Institute of Mining and Metallurgy
CRF	cemented rock fill
dmt	dry metric tonne
DTM	digital terrain model
EM	electromagnetic
EPA	Environmental Protection Agency
EVC’ s	Ecological Vegetation Classes
FAR	fresh air rise
FAusIMM	Fellow of The Australasian Institute of Mining and Metallurgy
GDip	Graduate Diploma
GEF	Gold and Exploration Finance Company of Australia
GMA/WMC	Gold Mines of Australia/Western Mining Corporation
GPS	global positioning system
GST	goods and services tax
g/t	grams per tonne
HBr	hydrobromic acid
HCl	hydrochloric acid

HR	hydraulic radius
Hwy	Highway
ICP - AES	inductively couple plasma atomic emission spectroscopy
ID^2	inverse distance squared
ID^3	inverse distance cubed
IP	induced polarisation

Abbreviation	Meaning
IRR	internal rate of return
JORC	Joint Ore Reserves Committee
kg	kilogram
kL	kilolitre
km	kilometer
koz	kilo ounces
kt	kilotonne
ktpa	kilotonnes per annum
ktpm	kilotonnes per month
kV	kilovolt
kVA	kilovolt ampere
kW	kilowatt
kWh	kilowatt hour
L	litres
LHD	load-haul -dump
LOM/ LoM	life of mine
L/s	litres per second
M	million
Ma	million years
Newmarket Gold	Newmarket Gold Inc
MAusIMM(CP)	Member of The Australasian Institute of Mining and Metallurgy
mg/kg	milligrams per kilogram
mg/L	milligrams per litre
mH	meters high
ML	million litres
mm	millimeters
MMI	mobile metal ion
Moz	million ounces
mRL	meters reduced level
MRSD Act	Mineral Resources (Sustainable Development) Act 1990
Mtpa	million tonnes per annum
m^3	cubic meters
m^3/s	cubic meter per second
$m^3/s/KW$	cubic meter per second per kilowatt
MVA	megawatt ampere
mW	meters wide

MW	megawatt
NI 43-101	National Instrument 43-101
NPV	net present value
OH & S	Occupational Health and Safety
Onsite	Onsite Laboratory Services
ozs	ounces
PEA	Preliminary Economic Assessment
PhD	Doctor of Philosophy
Planet	Planet Resources Group NL
POX	Pressure oxidation

Abbreviation	Meaning
QA/QC	quality assurance/quality control
QP	Qualified Person
RAR	return air raise
RC	reverse circulation
ROM	run-of -mine
SD	standard deviation
SRK	SRK Consulting (Australasia) Pty Ltd
t	tonnes
tpa	tonnes per annum
t/mth	tonnes per month
TSF	tailings storage facility
TSX	Toronto Stock Exchange
UCS	unconfined compressive strength
USD	US dollars
V	volt

Executive Summary

Introduction

SRK Consulting (Australasia) Pty Ltd (SRK) was engaged by Newmarket Gold to undertake a study on the Maud Creek Gold Project (Maud Creek or the Project) and prepare a Technical Report summarising the findings of a Preliminary Economic Assessment (PEA).

In early July 2015, Newmarket Gold Inc. merged with Crocodile Gold Corp. (Crocodile Gold) to form a new Canadian, Toronto Stock Exchange listed gold mining company named Newmarket Gold that has 100% ownership of the Maud Creek Project.

The work and this Technical Report were prepared by SRK following the guidelines of the Canadian Securities Administrators' NI 43-101 and Form 43-101 F1.

The Mineral Resource statement reported herein was prepared in conformity with generally accepted Canadian Institute of Mining, Metallurgy, and Petroleum' s (CIM) Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines.

Scope

The scope of this Technical Report documents the findings of the PEA that included:

- an assessment of the project' s geology and exploration leading to a release of a Mineral Resource estimate;
- an assessment of the geotechnical data and recommendation for geotechnical design guidelines;
- an preparation of open pit and underground mine designs and schedules;
- an assessment of the metallurgical considerations and preparation of plant design and metallurgical performance assumptions;
- an assessment of the hydrogeology and hydrological aspects;
- an assessment of the infrastructure requirements;
- an assessment of the environmental and permitting considerations;
- an estimate of operating and capital cost inputs; and
- techno-economic modelling.

SRK' s scope included consideration for processing mineralization at Newmarket' s' Union Reefs Processing Plant (including oxide mineralization) and through a stand-alone plant constructed on-site at Maud Creek (excluding oxide mineralization) to produce a saleable gold concentrate and gold Dore.

A previous Technical Report, Bremner, P and Edwards, M, 2012. Report on the Mineral Resource and Mineral Reserve of the Maud Creek Gold Project, excluded Oxide mineralization, assumed processing including a Bacterial Oxidation plant (BIOX®) and sale of gold Dore.

This Technical Report summarises the findings of the assessment of the option of processing mineralization at the Union Reefs Processing Plant. The techno-economic results of the PEA are presented in Table ES-1

Table ES-1: Summary of PEA Results

Parameter/ Result	Units	Quantity
Gold Price	AUD/oz	1,550
Exchange Rate	AUD: USD	0.77
Gold Price	USD/oz	1,200
Mine Life	Years	9.5
Mineral Inventory	'000 t	3,911
Diluted Gold Grade	g/t	4.2
Contained Gold	koz	528
Gold Recovery (oxide/transitional)	%	85
Gold Recovery (sulphide)	%	95
LOM Recovered Gold	koz	496
Production Rate	ktpa	500
Average Annual Gold Production	koz	52
Peak Annual Gold Production	koz	70
Annual Tonnes Concentrate (Dry)	kt	30
Concentrate Grade	g/t con	45
LOM Operating Cost	AUDM	408
LOM Cash Operating Cost	AUD/oz	1,101
Total Operating Costs/tonne Milled	AUD/t	105
Net Revenue (less selling expenses)	AUDM	725
Pre-Production Capital cost	AUDM	42
Sustaining Capital Cost (LOM)	AUDM	14

It is important to note that the PEA is preliminary in nature that it includes Inferred mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and there is certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have documented economic viability.

Based on the results of the comparisons of the economics, Table ES-2, and infrastructure considerations associated with both options the decision was taken to present the option of utilising the Union Reefs Processing Plant as the preferred option.

The economic analysis treats the Northern Territory Royalty as a Post-Tax cost, as is it is influenced by the NT Royalty Net Negative Value. SRK notes that Newmarket Gold NT Holdings Pty Ltd carries tax losses that have not been included in the economic analysis. Based on the integration of the Project this has the potential to further impact the value of the Project. The Northern Territory Royalty with no "losses carried forward" is $107 per ounce. The estimated available tax losses and NT Royalty Net Negative Value as at 31 December 2015, provided by Newmarket Gold is:

Income tax non-capital losses AUD229.8M

NT Royalty Net Negative Value AUD151.0M.

Table ES-2: Comparison of Options

	Units	Union Reefs Plant Quantity	Stand-alone Plant Quantity
Mine Life	years	9.5	8.0
Mineral Inventory	'000 t	3,911	3,460
Gold Grade	g/t	4.2	4.1
Contained Gold	koz	528	458
LOM Recovered Gold	koz	496	433
Gold Price	AUD/oz	1,550	1,550
Exchange Rate	AUD: USD	0.77	0.77
Gold Price	USD/oz	1,200	1,200
LOM operating cost	AUDM	408	378
Net Revenue (less selling expenses)	AUDM	725	633
Capital cost	AUD M	56	121
Pre-tax Cashflow	AUDM	261	134
Pre-tax NPV (5)	AUDM	201	89
After Tax Cash Flow	AUDM	182	91
After Tax NPV (5)	AUDM	137	55

Property Description and Location

The Maud Creek Gold project (Maud Creek or the Project) is located within the Pine Creek region of the Northern Territory of Australia, 20 kilometers east of Katherine. Previous mining activities at Maud Creek have been limited to open pit mining during 2000 when the owner was AngloGold.

The project comprises a total of 4 mineral titles (all granted), and the deposit is located wholly within tenement ML30260 which is held 100% by Newmarket Gold. Maud Creek is located at latitude 14°26' 41" south and longitude 132°27' 10" east.

Accessibility, Climate, Local Resources, Infrastructure and Physiography

Access is gained to the Project from Darwin by travelling south for 314 road kilometers along the sealed Stuart Highway to the town of Katherine.

Darwin has a population in excess of 129,000 and is the capital city of the Northern Territory. It is the administrative centre of the Northern Territory government and a major transportation hub, with an international airport and deep water port and the Adelaide to Darwin transcontinental railway terminating at the East Arm port.

Katherine is a regional centre with a population of approximately 9,800 and enjoys excellent infrastructure, services and communications. This is the closest centre of population to the Maud Creek project. Nearby regional mining communities of Pine Creek (with a population of 450) and Adelaide River (population of 200) support the Burnside, Maud Creek and Moline gold projects.

The major land use is grazing on native pastures and traditional Indigenous uses with some horticulture, grazing on modified pastures and nature conservation. The region has undergone some clearing (approximately 167,000 ha) for these developments The vegetation of the Maud Creek area consists largely of woodlands and open woodlands (predominant species - Eucalypts) that have been degraded by the impacts of cattle, buffalo and wild donkeys. No rare, threatened or endangered species have been identified in the area.

In the Maud Creek area, the terrain is flat lying to undulating. Ephemeral streams transect the project area and drain into the westward flowing Katherine River that flows all year.

The Top End of the Northern Territory has a tropical monsoon climate characterized by two distinct seasonal patterns: the ‘wet’ monsoon and the ‘dry’ seasons. The wet season generally occurs from November through to April and the dry season between May and October. Almost all rainfall occurs during the wet season, mostly between December and March, and the total rainfall decreases with distance from the coast.

History

Gold was initially discovered in the Maud Creek area in 1890 and a small plant was set up but ultimately abandoned in 1891. This is now called the Chlorite Hills and O’ Shea’ s area.

The area was re looked at from 1932 34 when 400 tonnes of ore produced 540 ounces of gold. Mining was from about 20 shallow shafts and small holes that were 6 12 meters deep with horizontal workings from 15 30 meters in length in the Chlorite Hills and O’ Shea’ s area.

Interest in the Maud Creek area was rekindled in the 1960s during an assessment of the mineral potential of the Top End of the Northern Territory. This study was prompted by the discovery of significant uranium mineralization in the nearby South Alligator River valley in the mid 1950s.

The Maud Creek project was owned by a number of companies until the acquisition of the Project by GBS Gold in December 2006. Substantial drilling, in the order of 66,000 90,000 m of RC and diamond drilling, is reported on the Project area during the period 1966 – 2006, oriented toward gold exploration.

AngloGold acquired rights to mine the oxide zone of the Main Zone deposit at Maud Creek and be processed at the Union Reefs Processing Plant. Mining operations were conducted during 2000. A total of 173,581 tonnes at 3.32 g/t Au produced 18,527 ounces. Ore was trucked from Maud Creek to the Union Reefs Processing Plant.

An agreement to acquire a number of properties, including the Maud Creek property, was entered into on June 19 2009 from GBS Gold International Inc. (GBS Gold) (In liquidation). GBS Gold operated the Tom’ s Gully and Brock’ s Creek underground gold mines, mined several open pit gold deposits and operated two gold Processing Plants, one at Tom’ s Gully, the other at Union Reefs, near Pine Creek, Northern Territory, until September 2008, when administrators were appointed.

On November 6 2009, the mining tenements including the Maud Creek Property were registered in the name of Crocodile Gold Australia Pty Ltd, a subsidiary of Crocodile Gold, which became Newmarket Gold in July 2015.

Geological Setting and Mineralization

The Maud Creek Gold deposit is located in the south-eastern part of the Pine Creek Geosyncline, within the Gold Creek Fault Zone, which forms the contact between mafic tuffs of the Dorothy Volcanics to the east and sedimentary rocks of the Tollis Formation to the west.

The Tollis Formation is the youngest member of the Finniss River Group, has limited aerial extent and consists of a succession of interbedded mudstone, slate, metagreywacke and minor felsic volcaniclastic shales. The Dorothy Volcanic Member consists of volcanic tuff with minor interbedded zones of sediments.

The north-south trending Gold Creek Fault Zone and primary Maud Creek mineralized zones dip steeply to the east. The deposit is roughly bound to the east by the Maud Creek Dolerite, which also exhibits mineralization at the tuff/dolerite contact and to the north by a small andesite body located at the contact between the sandstone and tuff (Maud Creek Contact Fault). To the south of the deposit a major east-west structure with sinistral strike-slip movement has been interpreted. Eight faults have been identified in the Maud Creek deposit area and generally exhibit reverse movement, with limited offsets in the range of meters.

The Maud Creek Contact Fault is filled with quartz stockwork veins; three vein lodes have been modelled for the Maud Creek deposit; the primary contact vein, upper contact vein and lower contact vein. The primary vein is strongly associated with the sandstone/tuff contact but does not strictly follow the boundary. Therefore, it has been modelled as an 'overprinting' volume onto the sediment, tuff, dolerite and andesite lithology wireframes. Mineralization in the east at the tuff/dolerite contact generally form steeply dipping discrete lenses with limited continuity.

Outside of and adjacent to the Maud Creek Contact Fault mineralization are many intercepts carrying similar grades to those within the vein itself; these extend up to 25 meters into the hanging wall and to a lesser extent into the footwall. In addition, a greater than 0.1 g/t Au halo can be observed up to 50 meters into the hanging wall and occasionally in the footwall.

Deposit Types

A variety of genetic models have been postulated for the formation of gold deposits in the Pine Creek Geosyncline. Gold and base metal mineralization is commonly associated with granite intrusions and are often been classified as high temperature contact aureole deposits. A secondary host rock control has also been suggested due to the association of gold mineralization with carbonaceous metasedimentary rocks. More recently, authors have argued that gold mineralization is structurally controlled; occurring in brittle ductile structures at the greenschist amphibole facies boundary and hence has an epigenetic origin.

Accepting that gold deposits of the Northern Territory have a structurally controlled mesothermal setting, then on the basis of host rock and mineral association they can be divided into seven types:

Gold quartz veins, lodes, sheeted veins, stockworks, saddle reefs (Pine Creek Orogen);

Gold ironstone bodies (Tennant Inlier);

Gold in iron rich sediments (Pine Creek Orogen, Tanami);

Polymetallic deposits (Iron Blow, Mt Bonnie);

Gold PGE deposits (South Alligator River area);

Uranium gold deposits (Pine Creek Orogen, Murphy Inlier); and

Placer deposits.

Of these types, Maud Creek aligns with the gold-quartz veins, lodes, sheeted veins, stockwork deposit type. Five main types of mineralization have previously been recognized within the Pine Creek Orogen. These include:

Sheeted and stockwork quartz vein systems located along major anticlinal hinges;

Sediment hosted stratiform gold mineralization and quartz sulphide vein hosted stratabound

gold mineralization in cherty ironstone and carbonaceous mudstone;

Stratiform, massive to banded, sulphide silicate carbonate mineralization ;

Sediment hosted stratiform and stratabound gold mineralization in cherty, dolomitic and

sulphidic shales; and

Sheeted or stockwork quartz feldspar sulphide veins.

Of these mineralization types, Maud Creek is consistent with stockwork quartz-feldspar-sulphide veining hosted at the contact of either sandstone/tuff or tuff/dolerite units.

Mineral Resource Estimates

The Mineral Resources are stated here for the Maud Creek deposit with an effective date of 15 March 2016 and were previously reported by Fairfield and Kentwell (2016).

The Maud Creek deposit consists of open pit and underground resources presented in Table ES-3 and ES-4. All relevant diamond drillhole samples, available as of April 2015 for the Maud Creek deposit were used to inform the estimate. The estimation methodology utilised was Ordinary Kriging (OK) to estimate gold and arsenic using hard domain boundaries.

Table ES-3: Open pit Mineral Resource above 950 mRL at 0.5 g/t Au cut-off – base case

Mineral Resource Category	Inventory (kt)	Gold Grade (g/t)	Contained Metal (koz Au)
Measured	1,070	5.6	190
Indicated	1,100	2.1	75
Measured and Indicated	**2,170**	**3.8**	**268**
Inferred	530	1.4	25

It should be pointed out the Mineral Resource estimate is categorized as Measured, Indicated and Inferred as defined by the CIM guidelines for resource reporting. Mineral resources do not demonstrate economic viability, and there is no certainty that these Mineral Resources will be converted into mineable reserves once economic considerations are applied. The Measured, Indicated and Inferred Mineral Resource estimate has been prepared in compliance with the standards of NI 43 – 101 by Danny Kentwell, FAusIMM.

Notes to Table ES-3:

1. CIM definitions followed for classification of Measured, Indicated, and Inferred Mineral Resources.
2. Mineral Resources estimated as of 15 March 2016.
3. Mineral Resources stated according to CIM guidelines.
4. Totals may appear different from the sum of their components due to rounding.
5. Reported at a 0.5 g/t cut-off grade.
6. The open pit Mineral Resource is exclusive of the underground Mineral Resource.
7. The Mineral Resource estimation was performed by Danny Kentwell FAusIMM fulltime employee of SRK Consulting, who is a Qualified Person under NI 43-101.

Table ES- 4: Underground Mineral Resource below 950 mRL at 1.5 g/t Au cut-off – base case

Mineral Resource Category	Inventory (kt)	Gold Grade (g/t)	Contained Metal (koz Au)
Measured	-	-	-
Indicated	4,330	3.2	456
Measured and Indicated	**4,330**	**3.2**	**456**
Inferred	1,450	2.7	124

It should be pointed out the Mineral Resource estimate is categorized as Indicated and Inferred as defined by the CIM guidelines for resource reporting. Mineral resources do not demonstrate economic viability, and there is no certainty that these Mineral Resources will be converted into mineable reserves once economic considerations are applied. The Measured, Indicated and Inferred Mineral Resource estimate has been prepared in compliance with the standards of NI 43-101 by Danny Kentwell, FAusIMM.

Notes to Table ES-4:

1. CIM definitions followed for classification of Measured, Indicated, and Inferred Mineral Resources.
2. Mineral Resources estimated as of 15 March 2016.
3. Mineral Resources stated according to CIM guidelines.
4. Totals may appear different from the sum of their components due to rounding.
5. Reported at a 1.5 g/t cut-off grade.
6. The underground Mineral Resource is exclusive of the open pit Mineral Resource.
7. The Mineral Resource estimation was performed by Danny Kentwell FAusIMM fulltime employee of SRK Consulting, who is a Qualified Person under NI 43-101.

In SRK' s opinion, based on the depth and distribution of the mineralization open pit and underground mining could be viable options for extraction.

In assessing the criteria for reasonable prospects of economic extraction both open pit and underground scenarios were considered. With respect the scattered lower grade mineralization contained within the near surface a simple pit optimisation using the optimistic parameters at twice the current gold spot price did not generate a pit of practical size on the eastern domains. All material in the eastern domains is not considered to have reasonable prospects of economic extraction and does not appear in the Mineral Resource.

With respect to the underground potential the grade is reasonably consistent down to approximately 650 mRL below which it drops significantly. All material below 650 mRL is not considered to have reasonable prospects of economic extraction and does not appear in the Resource.

Mining

Based on the geological review of the deposit, SRK has prepared a mining design and schedule based on a combination of conventional open pit and underground mining operations. The proposed mill feed estimated from SRK' s design is presented in Table ES-5 and a breakdown by classification in Table ES-6.

Table ES- 5: Proposed Mill Feed

	Inventory (kt)	Grade (g/t)	Contained Metal (koz Au)
Open Pit	634	5.1	104
Underground	3,276	4.0	423
Total	**3,911**	**4.2**	**528**

Table ES-6: Underground Mining Inventory by Resource Classification

Resource Classification	% of Feed	Diluted Mined Tonnes and Grades		
		Inventory (kt)	Grade (g/t Au)	Contained Metal (koz Au)
Measured	20	679	6.1	132
Indicated	70	2,306	3.9	290
Inferred	10	291	2.8	26
Underground Mining Inventory	**100**	**3,276**	**4.0**	**423**

It is important to note that the PEA is preliminary in nature that it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and there is certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have documented economic viability.

Geotechnical

In conjunction with the Mineral Resource review, a comprehensive review of the available data and analysis to determine open pit and underground geotechnical design guidelines.

An assessment of overall slope angles and underground mining parameters has been undertaken using geological and geotechnical drilling data supplied by Newmarket Gold. The analysis provides good early-stage design guidelines of the geotechnical properties of the rock mass. The typical geotechnical conditions on site can be summarised as follows:

> The Hanging wall Tuffs are typically massive but may be locally bedded. Hanging wall tuffs are also affected by the numerous shears present in the Hanging wall, resulting in reduced strength, increased fracture frequency and graphitic and/or chloritic alteration of the rock mass.
>
> The Footwall Sediments consist of low to medium strength thinly bedded or laminated mudstone and siltstone, and medium to thickly bedded sandstone. Zones of intense shearing with chlorite and graphite alteration occurring in the 5 to 10 meters below the mineralized zone where the sediments are commonly black, highly graphitic and/or chloritic, very weak and fissile.
>
> The competency of the mineralized zone can be expected to be variable with competent, partially silicified mineralized zones separated by zones of intensely sheared rock.
>
> The distribution of the various fault configurations is not understood at this stage and this should be one of the main focus for subsequent field investigations.

The absence of suitable data has led to low-confidence in the geotechnical conditions. Additional data is required to improve confidence and refine decisions on mining methods and the mine design. The mining method studies are linked to the decision on the location of the Processing Plant and the availability of pastefill.

The proposed base case is utilisation of the Processing Plant being located at Union Reefs; as such a waste fill mining option has been considered with a “bottom-up” mining sequence.

Hydrogeology

SRK reviewed the findings and basis of previous reports and considers that them to be fit for purpose. Based on the absence of the underlying data, the purpose and stage of this study it was not appropriate to undertake additional hydrogeological modelling.

Should the study progress, additional data is required and should be incorporated with the revised structural / geological model to develop an updated numerical model. This would either build on the existing model, if data can be made available or a new model would be constructed. Specifically the updated model would confirm / address dewatering rate assumptions and would seek to collect data at depth. There is a distinct lack of knowledge at depth in the existing model.

Water management of the site has been incorporated as a key factor, to keep clean water clean and direct the contact water to appropriate containment systems. A preliminary analysis has been completed to prevent flows associated with the 1-in-1000 year ARI event from encroaching the open pit.

The designed infrastructure includes a flood protection bund to the east of the pit, a diversion channel to maintain Gold Creek to the east of the pit has been conceptually designed and a dedicated water pond for the excess water.

Based on the available data, the preliminary water balance indicates the water demand is met throughout operations. There is excess water in the system to be treated and managed on-site. SRK has modelled the use of a water treatment plant as opposed to an evaporation facility.

Additional study work will be required once a greater understanding of the water quality and quantity is determined.

Mineral Processing & Metallurgical Testwork

An extensive program of metallurgical testing was carried out from 1994 through to 2006 at reputable and suitably experienced laboratories. Testing was undertaken at both batch and pilot scale and including variability testing. Part of the focus of testing was on downstream oxidation processes on the refractory and preg-robbing Maud Creek mineralization, such as bio-oxidation (BIOX®) and the GEOCOAT® process. Direct cyanidation leaching of mineralization and concentrates was tested on the fresh (sulphide) mineralization with poor results and was eliminated as a potential processing route. A number of engineering studies were undertaken in conjunction with this testwork.

Metallurgical testing has shown the fresh mineralization, the bulk of the Project tonnage, to be moderately hard and abrasive, to have variable levels of gravity gold recovery, to be refractory and preg-robbing in nature but responsive to simple flotation techniques - demonstrating high gold recoveries in excess of 95%. Total recovery is consistently high irrespective of gravity recovery. The flotation concentrate has sufficient grade to be classified as a gold concentrate for the purposes of importation into China (> 40 g/t). It is noted that part of the gold is associated with arsenopyrite and as a result, arsenic grades in the concentrate are elevated at approximately 3.6% . There is no arsenic restriction on the importation of gold concentrates.

The Maud Creek mineralization has been subject to extensive metallurgical testing, adequate to support this PEA. Some additional testwork is recommended by SRK for the next level of design to further increase confidence of the study if new drill core becomes available. This includes variability testing of metallurgical behaviour at depth (gravity and comminution particularly), flash flotation sighter tests and potentially some further assessment of the oxide and transitional mineralization. The customers will likely require a sample of the final flotation concentrate for their own testing. This will necessitate additional metallurgical testwork at some point and can be done in conjunction with the other recommended testing requirements. Some assessment of reagent contamination of the flotation circuit (specifically cyanide) and the impact on flotation recovery is also important now that the Project base case assumes processing is at Union Reefs.

Maud Creek oxide mineralization can be processed through the existing Union Reefs CIP plant and can be suitable treated. The transitional feed, which makes up just a small portion of the overall LoM tonnage but a significant part of the first years feed will have more variable metallurgical behaviour. Transitional mineralization can be processed through the existing oxide circuit and/or the new flotation circuit to optimize recovery. Sulphidation flotation testing is a possibility to optimize transitional mineralization recovery.

The current PEA has reviewed the previous metallurgical testwork data and supporting engineering studies. From this body of work, a simple, conventional, inexpensive and low-risk processing circuit has been selected, allowing flexibility with respect to circuit configuration and downstream processing. It originally assumed a standalone Maud Creek Processing Plant processing fresh mineralization only. The engineering undertaken as part of this PEA reflects this. Subsequently the Project assumption changed to processing mineralization through a modified Union Reefs Processing Plant as the base case. Any preliminary engineering gaps are largely associated with this change.

The base case presented for this study is a Brownfield upgrade to the existing and operating Union Reefs Processing Plant. This would see the addition of flotation and concentrate dewatering circuits, producing a gold sulphide concentrate for sale. Part of the concentrate production could be processed at Newmarket Gold' s Fosterville BIOX® plant if spare capacity is available. A gravity gold concentrate will also be produced. The preference would be to smelt it into gold doré on site but alternatively it could be smelted and refined in Australia at one of the established gold refiners.

A detailed integration study has not been undertaken into the Union Reefs processing option. This will be undertaken at the next Phase of the Study. However it is expected to be relatively simple, quick and straight forward. It presents a significant opportunity to processing of Maud Creek mineralization.

At the Union Reefs Processing Plant, the ore is crushed and then milled in closed circuit with hydrocyclones. A separate gravity circuit recovers gravity recoverable gold from the ball mill discharge. The fine milled product from the cyclone overflow is pumped to a new flotation circuit where a high grade low tonnage gold bearing concentrate is produced from the sulphide minerals, leaving a gangue non-sulphide residue that can be disposed of into tailings. The concentrate is dewatered through thickening and filtration before being bagged, stored in shipping containers and transported by road, then ship to customers in China.

Downstream processing options for refractory gold mineralization and concentrates were considered at a preliminary level. The direct smelting of flotation concentrates option has been selected as the preferred option.

There are a number of advantages associated with leveraging the existing Union Reefs Processing Plant rather than building a standalone Processing Plant at Maud Creek including;

- Lower processing and infrastructure capital costs;
- Lower first fill costs;
- Low additional sustaining capital costs;
- Simpler and quicker project implementation, reduced technical risk;
- Lower water demand at Maud Creek;
- Less onerous approvals for existing processing facility;
- Lower overall operating cost;
- Production creep opportunities with existing facility;
- Simple and cheap future expansion to meet any increase in mining production;
- Ability to process oxide mineralization, so increasing the LoM of Maud Creek;
- More flexibility in processing, including transitional mineralization being able to be processed through the cyanide leach or flotation circuits or possible both, (higher gold recovery);
- Extensive tailings storage capacity already available (lower capital cost); and
- Extend the LoM of the Union Reefs/Cosmo underground mine operation.

The key additional cost is haulage of mineralization. Other key considerations include the required approvals required for this haulage and obtaining the social license to operate.

Project Infrastructure

The Maud Creek Gold Project surface facilities are representative of a modern and conventional underground mining operation. The site comprises the following:

- Office and administration complex, including change house;
- Store and laydown facilities;
- Heavy underground equipment workshop;
- Temporary surface stockpiles and waste stockpile area;

Maud Creek Open Pit Mine and portal;

Ventilation exhaust raises;

Ventilation intake raise;

Raw water storage to manage rainfall runoff; and

ROM haul road along the existing access road to the town of Katherine.

The Union Reefs Processing Plant was commissioned in 1994 with an upgrade in 1998. Key infrastructure at the plant includes:

Office and administration complex;

Store and laydown facilities;

Gravity and CIL Processing Plant and associated facilities;

Process plant workshop;

Electrical substations and transformer supplied by 66 kV power line;

Reagent storage;

Laboratory;

Vehicle wash-down area;

Tailings storage facilities;

Run of Mine stockpiles; and

Core processing facility.

The processing team will continue to be provided from the Cosmo Underground Mine accommodation village located 53 km north of Union Reefs, or through residential employment based in the nearby towns of Pine Creek, Adelaide River and Katherine. The Cosmo Accommodation Village is managed by an independent contractor. Maud Creek mining accommodation will be provided for in Katherine or the local surrounds.

Ore Haulage

ROM mineralization will be hauled 144 km from the Maud Creek Mine to the existing Union Reefs Processing Plant. The access road form Katherine to Maud Creek will be upgraded to allow for heavy haulage. A preliminary study into light and heavy vehicle movements on the proposed haul route in and around Katherine has been generated using publicly available 2014 vehicle movements on the Stuart Hwy.

It is anticipated that the mine haulage operations will utilise quad semi-trailers, which will involve travel from Maud Creek to Union Reefs (loaded) and back again (empty) on a daily basis; therefore, the mining operations will require approximately 26 road train movements per day (13 full and 13 empty). It is proposed that haulage from the site and traffic to the site will occur only during daylight hours. This would nominally be from 6:00AM to 6:00PM. This will occur throughout the year during both the wet and dry seasons. Confirmation of allowable number of trailers and load through town centres is required at the next phase of the pre-feasibility study. Any reduction will require a corresponding increase in truck movements each day.

The increase in traffic on the Stuart Highway resulting from the mining operations at Maud Creek would be 2.4% north of Katherine, 6.9 percent south of Katherine and 0.7% within the Katherine Township. More significantly, in terms of heavy vehicle movements, the 26 quad road train movements per day represents a 24.6% increase north of Katherine and a 26.9% south of Katherine.

The major safety impacts of these road trains will be at the two intersections where the road trains turn onto or off the Stuart Highway and ensuring adequate overtaking opportunities for cars.

In July 2015, the Northern Territory Government started the planning on a long-term heavy vehicle alternate route that bypasses the central business area of the township. The alternate heavy vehicle route study was proposed following a number of road safety issues in the main street, as it currently accommodates a mix of pedestrians, local and tourist traffic, and heavy vehicles. The alternate heavy vehicle route would dramatically reduce the impact of the Maud Creek haulage operations. Government and public consultation and approval will be a key element to the viability of processing of Maud Creek mineralization at the Union Reefs Processing Plant.

Power

A preliminary power supply options assessment has been undertaken to supply the approximately 2.0 MW of power demand for the underground mining operations. It considered mains power and site generated power (diesel and/or natural gas). Due to the limited demand from mining and ramp up in requirement, it is proposed that electrical power is provided via on site diesel generators supplied, owned and operated by an independent power provider (IPP).

Power at the Union Reefs Processing Plant is supplied via the 66 kV Darwin-Katherine distribution network. There is sufficient capacity to meet the modifications to the plant. The difference in power demand operating the existing CIL circuit or the new flotation circuit would not be significant.

Concentrate Transport & Contracts

The PEA identifies the potential for 36,500 tonnes (dry) of gold concentrate to be produced annually from the Maud Creek deposit. The concentrate will be stored in 1.5 tonne bulk bags which are then loaded into shipping containers ready for transportation. Containers are trucked from the Union Reefs Processing Plant to the Darwin Port, approximately 220 km. From there they are shipped to Chinese customers. At Darwin Port, the containers are stacked until required for ship loading with fortnightly sailings scheduled. Preliminary costs assume Dalian Port as the destination port in China. This has a transit time of 29-36 days via Shanghai or Kaoshiung.

Indicative terms have been provided to Simulus Engineers for the sale of a gold concentrate to Shangdong Zhong Guo (China Gold Shandong) for processing at their Yantai Gold Smelter. Payment terms are USD10/t for processing and payment for 95% of the contained gold. Similar terms were provided by Baxville (Beijing) Minerals Trading Ltd. Additional smelting terms were provided verbally by Australian gold concentrate producers to allow further benchmarking.

Other inquiries have been issued to potential customers and traders. Indicative terms for the sale to another direct exporter of concentrate (i.e. to consolidate with their concentrate) who sell to Guoda Gold Co Ltd, or sale to an Australasian Pressure Oxidation (POX) facility with additional capacity have been referenced from other similar gold concentrates projects and operations in Australia. Informal discussions have also been held with the owners of a number of Australian third party refractory gold operations for sale of a generic gold concentrate and to assess available capacity at those facilities but discussions have not been progressed to any great extent. It has been used mainly in the process of elimination of the bulk of these options. Further investigations into the remaining options can be undertaken at the next stage of study if deemed appropriate.

The sale of the gold concentrate to China remains the current base case. No formal concentrate discussions or sales contracts have been entered into at this stage of study. Gravity gold will be recovered into gold doré on site and sold to established Australian refiners.

Economic Analysis

The key economic assumptions are provided in Table ES-7. A summary of the project economics are presented in Table ES-8. SRK notes the gold price at the time of reporting is over AUD1,600/oz. SRK has applied a discount rate of 5% on the basis that Newmarket Gold is a Canadian based company and will source funding from US/ Canadian markets. An increase in the discount rate to 8% reduces the After Tax NPV by AUD20M.

It is important to note that the PEA is preliminary in nature that it includes Inferred mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and there is certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have documented economic viability.

Newmarket Gold NT Holdings Pty Ltd carries tax losses that have not been included in the economic analysis. The estimated available tax losses as at 31 December 2015, provided by Newmarket Gold is:

Income tax non-capital losses AUD229.8M

NT Royalty Net Negative Value AUD151.0M.

Table ES-7: Economic Assumption Criteria

Description	Units	Quantity
Gold Price	Gold AUD/oz	1,550
Exchange Rate	AUD: USD	0.77
Gold Price	USD/oz	1,200
Discount Rate	%	5

Table ES-8: Project Economics

	Units	Union Reefs Plant Quantity	Stand-alone Plant Quantity
Mining			
Mine Life	years	9.5	8.0
Mineral Inventory	'000 t	3,911	3,460
Gold Grade	g/t	4.2	4.1
Contained Gold	koz	528	458
Open Pit Tonnes Ore	kt	634	168
Open Pit Gold Grade	g/t	5.12	6.1
Open pit Contained ounces	koz	104	33
Total Waste Mined	'000 t	5,000	1,358
Strip Ratio (O/P)	t:t	8	8
Underground Tonnes Ore	'000 t	3,276	3,292
Underground Gold Grade	g/t	4.02	4.02
Underground Contained ounces	koz	423	425
Processing			
LOM Tonnes Milled	'000 t	3,911	3,460
Production Rate	ktpa	500	500
Average Annual Gold production	koz	52	54

	Units	Union Reefs Plant Quantity	Stand-alone Plant Quantity
LOM Recovered Gold	koz	496	433
Metallurgical Recovery			
Transitional to Float	%	65	65
Transitional to Gravity	%	20	20
Fresh to Float	%	75	75
Fresh to Gravity	%	20	20
Annual Tonnes Concentrate (Dry)	kt	30	32
LOM Tonnes concentrate (Dry)	kt	285	253
Concentrate Grade	g/t con	45	45
Economics			
Gold Price	AUD/oz	1,550	1,550
Exchange Rate	AUD: USD	0.77	0.77
Gold Price	USD/oz	1,200	1,200
LOM operating cost	AUDM	408	378
LOM operating cost	AUD/t milled	105	109
LOM operating cost (Payable)	AUD/oz	856	910
LOM operating cost (Payable)	USD/oz	662	703
LOM Payable (Saleable Gold)	koz	477	416
Net Revenue (less selling expenses)	AUDM	725	633
Capital cost	AUD M	56	121
Pre-tax Cashflow	AUDM	261	134
Pre-tax NPV (5)	AUDM	201	89
Pre-tax IRR	%	116	28
Payback Period (pre-tax)	Qtr	6	15
After Tax Cash Flow	AUDM	182	91
After Tax NPV (5)	AUDM	137	55
After Tax IRR	%	80	20
Payback Period (After Tax)	Qtr	6	18

Sensitivity

The key drivers are the revenue assumptions (Grade, Recovery and Gold Price) and the operating cost assumptions.

SRK notes that Capital Cost estimates used for the sensitivity analysis exclude the capital development as the cost driver for this is the unit operating costs. Impact on the capital development costs is included in the operating cost sensitivity.

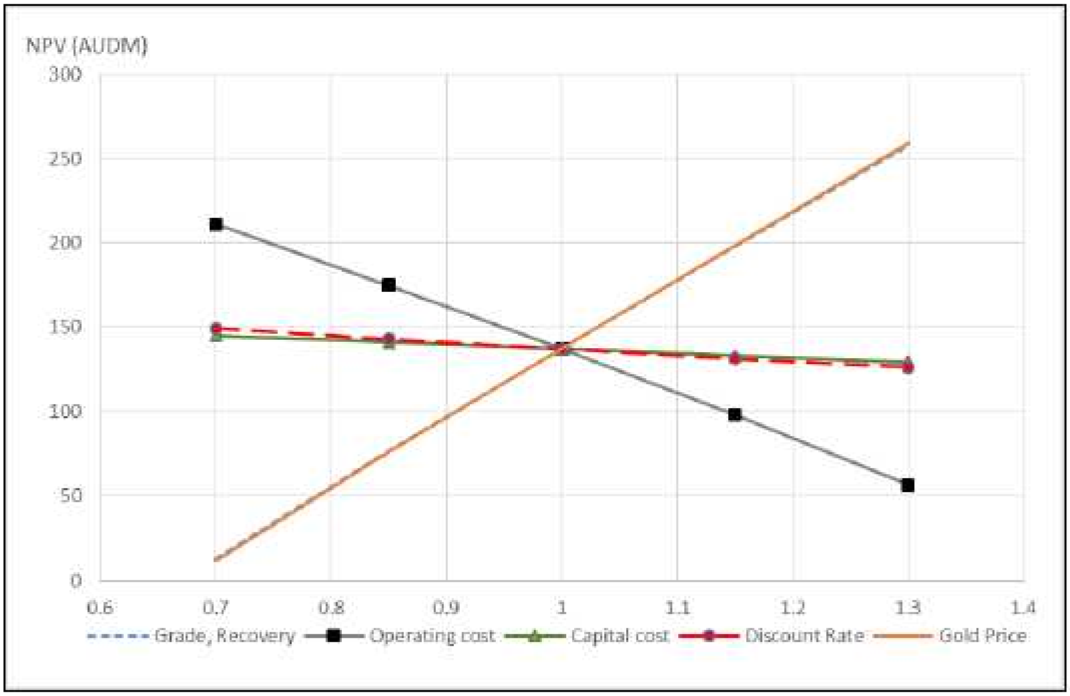

Figure ES-1: Sensitivity Analysis

Sensitivity to the Gold Price is presented in Table ES-9.

Table ES-9: Gold Price Sensitivity

Gold Price (AUD/oz)	**1,400**	**1,450**	**1,500**	**1,550**	**1,600**	**1,650**	**1,700**
Pre-Tax NPV5% (AUDM)	145	163	182	201	220	239	257
Pre-Tax IRR%	85	95	106	116	127	138	150
After-Tax NPV5% (AUDM)	98	111	124	137	150	163	177
After-Tax IRR%	59	66	73	80	87	94	102

Interpretation and Conclusions

It is important to note that the PEA is preliminary in nature that it includes Inferred mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and there is certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have documented economic viability.

Based on the economic findings of the PEA, SRK concludes that the Project has merit and that Newmarket Gold consider progressing the study to the next level of detail.

In doing so SRK provides recommendations that Newmarket Gold consider progressing the additional work programs to better understand the key technical risk areas to support future evaluations.

The 2015 geological model is considered by SRK to be more robust in terms of its geological basis and this has led to a slightly higher grades but a reduction in contained gold. Only further drilling can define true connectivity of the mineralization in widely spaced areas. Discussion of the risks and opportunities in the Mineral Resource model is presented in Table ES-10.

Table ES-10: Mineral Resource Model Risks and Opportunities

Project JAR Element	Economic Risk Level	Comment	Opportunity
Database - Exploration data	Low	Historical and recent data have been re- collated and re-validated for this Mineral Resource estimate.	
Assaying	Low	QAQC for recent and older assaying shows no material issues. Arsenic assaying has incomplete coverage.	Additional assaying for Arsenic may be beneficial depending on the processing method.
Surveying	Low	Both collar surveys and downhole surveys completed to a high level of accuracy for recent drilling. Representative collars resurveyed for older drilling with no significant discrepancies.	
Geology	Low	Detailed logging and interpretation together with evidence from both regional structural features and detailed in pit mapping informs the geological understanding	Additional drilling may be able to add detail to the interaction of structures controlling mineralization at depth.
Geological modelling	Low	A detailed structural and lithological model has been built and incorporated into the estimation domain construction.	Additional drilling may be able to add detail to the interaction of structures controlling mineralization at depth.
Resource Estimation	Low	Ordinary kriging cross checked and validated with theoretical grade tonnage curves and alternative search parameters has been used.	The project may benefit from simulation studies or non-linear estimates if detailed studies at selective mining unit block sizes are required in the future.

The interpreted level of confidence of the available Geotechnical data streams were rated subjectively using a 5-point rating scale of Very Low - Low - Moderate - High - Very High has been used (refer Table ES-11).

Table ES-11: Qualitative risk assessment of study components

Data	Confidence level
Empirical: rock mass characterisation	Low
Structural: major structures	Low to Moderate
Structural; rock mass structures	Low
Rock mass strengths	Very low
Rock material strengths	Very low
Groundwater conditions	Very low
Slope angle recommendations	Low

At this stage of the project development, a Low confidence rating should be expected for all items, with all items requiring further investigation. Aspects for which no data is currently available or represent a key concern have been flagged as Very Low.

The geotechnical assumptions are key inputs to the mine design and schedule and must be better understood prior to advancing the study work.

Recommendations

Should the Project study work proceed, SRK recommends the following aspects are considered:

Mineral Resource

Infill drilling in the parts of the Mineral Resource currently classified as Indicated and Inferred would enable an upgrade of the Mineral Resource Classification. An approximate drilling meters to complete this from surface down to 850 mRL is provided in Table ES-9 assuming the drilling takes place from surface.

A number of sections of the geological model remain open down dip with good grades seen in the last hole down dip. Extension drilling is recommended to test these areas. The required drilling meters to complete these are shown in Table ES-12 assuming drilling from surface.

Table ES-12: Recommended Mineral Resource Drilling

Target	Current exploration status	Potential End of 2016 Status	Description	Drilling (meters)
Infill Drilling	Indicated and/or Inferred	Measured and/or Indicated	Increase confidence in estimated Mineral Resource	9,200
Extension drilling	Down dip or along strike from current Mineral Resource	Indicated and/or Inferred	Close off or extend Mineral Resource volumes	2,200

Geotechnical

The geotechnical recommendations, summarised in Table ES-13, includes the following:

Dedicated geotechnical drilling program to:

- Target areas that require additional data such as the footwalls sediments, and crown pillar area;
- Drilling proposed locations of LOM infrastructure – portals, declines, vent shafts etc.;
- Improve understanding of structural characteristics including continuity and orientation variability;
- To collect representative samples for laboratory testing;

Mapping and photogrammetry in current pit;

Geotechnical laboratory testing using NATA certified laboratories;

Detailed backfill design for potential mining methods (including CRF);

Potential stress measurement testing (AE or DRA); and

Numerical modelling of proposed mining layout and sequence (including crown pillar and central pillar sequence)

Table ES-13: Drill program targets and estimated depths

Description	Drillholes (No.)	Depth (m)	Total (m)
To investigate sediments at depth	4	600	2,400
To supplement data and sediments across mid depths of mine	4	500	2,000
Portal and boxcut investigation drill holes	3	50	150
To investigate Infrastructure located in sediments (can be separated into multiple shorter drill holes)	1	600	600
Open Pit geotechnical holes – will provide data on crown pillar area and upper sediments	6	150	900
Total meters for proposed program	18		6,050

Geometallurgy

Insufficient information was available from the metallurgical reports to create preliminary spatial domains for the physical processing parameters, such as grindability. The metallurgical report Core Process Engineering Report No. 140-001 outlined JK Drop Tests which had been conducted, resulting in a Bond ball Work Index of 18-19 kWh/tonne for the main lode. Previous geology reports, as well as a site visit to inspect the core, gave an indication of the mineralogy of the Maud Creek deposit, which has an impact on the hardness of the mineralization. From a geometallurgical perspective, between and within each mineralized domain there will likely be a range of hardness values which will need to be established. Quartz alteration at Maud Creek has been identified as varying between the primary lode (high percentage quartz veining), moderate (footwall and hanging wall lodes stock work veining) to low (sandstone and tuff country rocks). However variations in silica content within these lodes will definitely occur, as alteration boundaries are normally pervasive across lithology boundaries. A possible way to better define the hardness parameters is the use of proxies. If silica analysis is included in any future assay testing of the mineralized zones, this can be used to identify target areas for JK drop weight tests. A regression calculation between the A*b result, Bond work indices and other comminution parameters versus the silica content can then be determined, allowing a predictive model of hardness to be created.

Mineral analysis would need to be conducted to ensure the silica content is reflecting quartz alteration which has a high hardness (Mohs scale 7) and not feldspar (Mohs scale 5-6) or mica minerals (Mohs scale 2-3) which have lower hardness and varying crystallography, resulting in different grinding behaviours. The Geology of the Maud Creek Gold Deposit and Maud Creek Reconciliation report by AngloGold in 2000 identified plagioclase laths and alkali feldspar minerals in the Andesite unit located to the south of the deposit, but not within the mineralized lodes. However, sericite (a type of muscovite) and chlorite alteration (most likely the mineral clinochlore) are both quite prevalent in all three mineralized zones, so their prevalence would need to be established as well. Similarly, the extent of the hematite and marcasite alteration noted in all three mineralized zones (present as pseudomorphs of pyrite in the oxide zone) needs to be quantified, as all three will have moderate-high hardness (6-7), but also a brittle tenacity upon breakage, which will influence the ore grindability.

The Maud Creek mineralization is going to be very hard and abrasive so finding a correlation with proxies such as silica, for example, is recommended by SRK. Equally, determining correlations between arsenic, sulphur and gold to help generate the flotation relationship correlations, and approximated the amount of arsenic in the final concentrate is also recommended. The arsenic and sulphur contents will also be a good indicator of recovery.

Mining

Should the project proceed SRK recommends that a range of sensitivities/ scenarios are considered at a high level to inform the project team of the implications of these constraints. The sensitivities should include combinations of considering the open pit oxide as an ore source with both the underground preferences retained and removed. Conventional open pit mining techniques are proposed.

The opportunity exists to modify the design to interact with the open pit design but this will potentially have constraints on production continuity.

The mine design and schedule should be revised based on the findings of the geotechnical review.

Processing & Infrastructure

The following additional recommendations are provided based on the findings from the first phase of study and the new base case being to process mineralization at the Union Reefs Processing Plant, after modifications, rather than at a standalone processing facility:

Undertake a further review of the metallurgical testwork to understand the behaviour of the oxide mineralization;

Undertake testwork to understand the intensive leach behaviour of the gravity gold concentrate, generate flotation concentrate for customer testing and assessment, other minor testwork as discussed in the main body of this report;

Undertake a specialist comminution circuit capacity study processing Maud Creek mineralization through Union Reefs Processing Plant;

Undertake a more detailed integration study of a new flotation, dewatering and concentrate storage circuits into the Union Reefs Processing Plant in consultation with the Union Reefs and Newmarket Gold technical groups. This would include scope of work, opportunity upgrades at Union Reef, tie-in requirements and the campaign operating philosophy;

Update the capital cost to suit the next level of study with Union Reefs as the base case;

Update the operating costs to suit the next level of study with Union Reefs as the base case, using their power, labour, accommodation and other costs. Agreement on the share of benefits to Maud Creek and Cosmo projects is also required;

Develop cost of upgrading the access road from the Maud Creek Mine Site into Katherine;

Progress discussions with the local Government authorities to confirm in principal that haulage through Katherine and up to Union Reefs is acceptable – this is a potential fatal flaw to the base case processing option;

Confirm capital upgrade requirements attributable to Newmarket Gold (if any) to the main Stuart Hwy; and

Confirm road train maximum tonnage acceptable to Local and Territory Government.

Progress flotation concentrate off-take discussions, to provide more confidence in terms and conditions.

Table of Contents

List of Tables

List of Figures

2 Introduction

SRK Consulting (Australasia) Pty Ltd (SRK) was engaged by Newmarket Gold to undertake a study on the Maud Creek Gold Project (Maud Creek or the Project) and prepare a Technical Report summarising the findings of a Preliminary Economic Assessment (PEA).

In early July 2015, Newmarket Gold Inc. merged with Crocodile Gold Corp. (Crocodile Gold) to form a new Canadian, Toronto Stock Exchange listed gold mining company named Newmarket Gold that has 100% ownership of the Maud Creek Project.

The work and this Technical Report were prepared by SRK following the guidelines of the Canadian Securities Administrators' NI 43-101 and Form 43-101 F1.

The Mineral Resource statement reported herein was prepared in conformity with generally accepted Canadian Institute of Mining, Metallurgy, and Petroleum' s (CIM) Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines.

2.1 Scope of Work

The scope of this Technical Report documents the findings of the PEA that included:

- an assessment of the project' s geology and exploration leading to a release of a Mineral Resource estimate;
- an assessment of the geotechnical data and recommendation for geotechnical design guidelines;
- an preparation of open pit and underground mine designs and schedules;
- an assessment of the metallurgical considerations and preparation of plant design and metallurgical performance assumptions;
- an assessment of the hydrogeology and hydrological aspects;
- an assessment of the infrastructure requirements;
- an assessment of the environmental and permitting considerations;
- an estimate of operating and capital cost inputs; and
- technical economic modelling.

SRK' s scope included consideration for processing mineralization at Newmarket' s' Union Reefs Processing Plant (including oxide mineralization) and through a stand-alone plant constructed on-site at Maud Creek (excluding oxide mineralization) to produce a saleable gold concentrate and gold Dore.

A previous Technical Report, Bremner, P and Edwards, M, 2012. Report on the Mineral Resource and Mineral Reserve of the Maud Creek Gold Project, excluded Oxide mineralization, assumed processing including a Bacterial Oxidation plant (BIOX®) and sale of gold Dore.

Based on the results of the comparisons of the economics, Table ES-1, and infrastructure considerations associated with both options the decision was taken to present the option of utilising the Union Reefs Processing Plant as the preferred option.

This Technical Report summarises the findings of the assessment of the option of processing mineralization at the Union Reefs processing Plant.

It is important to note that the PEA is preliminary in nature that it includes Inferred mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and there is certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have documented economic viability.

2.2 Work Program

The Technical Report was assembled in Melbourne during the months of April 2015 to May 2016. The Mineral Resource Statement reported herein was prepared in conformity with generally accepted CIM Exploration Best Practices and Estimation of Mineral Resource and Mineral Reserves Best Practices guidelines. This Technical Report was prepared following the guidelines of the Canadian Securities Administrators National Instrument 43-101 and Form 43-101 F1.

2.3 Basis of Technical Report

The purpose of this Technical Report is to present the geological review of the Maud Creek Gold Project. This report is based on information provided by Newmarket Gold to SRK and verified during site visits conducted in 2015 and any additional information provided by Newmarket Gold throughout the course of SRK' s investigations. The Qualified Persons have reviewed all relevant information and determined it to be adequate for the purposes of the Technical Report. The Qualified Persons do not disclaim any responsibility for this information. SRK has no reason to doubt the reliability of the information provided by Newmarket Gold.

This Technical Report is based on the following sources of information:

- Discussions with Newmarket Gold personnel;
- Inspection of Newmarket Gold' s Maud Creek Gold Project; and
- Additional information and studies provided by Newmarket Gold.

2.4 Qualifications of SRK and SRK Team

The SRK Group comprises over 1,400 professionals, offering expertise in a wide range of Resource engineering disciplines. The SRK Group' s independence is ensured by the fact that it holds no equity in any project and that its ownership rests solely with its staff. This fact permits SRK to provide its clients with conflict-free and objective recommendations on crucial judgment issues. SRK has a demonstrated track record in undertaking independent assessments of Mineral Resources and Mineral Reserves, project evaluations and audits, technical reports and independent feasibility evaluations to bankable standards on behalf of exploration and mining companies and financial institutions worldwide. The SRK Group has also worked with a large number of major international mining companies and their projects, providing mining industry consultancy service inputs.

The compilation of this Technical Report was completed by Peter Fairfield, Principal Consultant (Project Evaluation), BEng (Mining), FAusIMM (No 106754) CP (Mining). By virtue of his education, membership to a recognised professional association and relevant work experience, Peter Fairfield is an independent Qualified Person (QP) as defined by NI 43-101.

Danny Kentwell, Principal Consultant (Resource Evaluation), MSc Mathematics and Planning (Geostatistics), FAusIMM, undertook a review of the Mineral Resources and geological aspects of the project and contributed to the relevant sections in this Technical Report. By virtue of his education, membership to a recognised professional association and relevant work experience, Danny Kentwell is an independent QP as defined by NI 43-101.

Simon Walsh, SRK Associate Principal Metallurgist, BSc (Extractive Metallurgy & Chemistry), MBA Hons, CP, MAusIMM, GAICD undertook a review of the metallurgical, mineral processing and infrastructure aspects of the project. By virtue of his education, membership to a recognised professional association and relevant work experience is an independent QP as this term is defined by NI 43-101.

Table 2-1: Site Visits

QP	Position	Employer	Last Site Visit Date	Purpose of Visit
Peter Fairfield	Mining Principal Consultant	SRK	13 Aug 2014	Site Inspection
Simon Walsh	Processing Principal Consultant	Simulus	13 Aug 2014	Site Inspection
Rodney Brown	Geology Principal Consultant	SRK	13 Aug 2014	Site Inspection
Louie Human	Geotechnical Principal Consultant	SRK	3-7 Aug 2015	Geotechnical Logging
Tristan Cook	Geotechnical Consultant	SRK	3-7 Aug 2015	Geotechnical Logging
Kirsty Sheerin	Geology Consultant	SRK	3-7 Aug 2015	Geological Logging

2.5 Acknowledgement

SRK would like to acknowledge the support and collaboration provided by Newmarket Gold personnel for this assignment. Their collaboration was greatly appreciated and instrumental to the success of this project.

2.6 Declaration

SRK' s opinion contained herein is based on information collected by SRK throughout the course of SRK' s investigations, which in turn reflect various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results may be significantly more or less favourable.

This report may include technical information that requires subsequent calculations to derive sub-totals, totals and weighted averages.

Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material.

SRK is not an insider, associate or an affiliate of Newmarket Gold, and neither SRK nor any affiliate has acted as advisor to Newmarket Gold, its subsidiaries or its affiliates in connection with this project. The results of the technical review by SRK are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings.

3 Reliance on Other Experts

This report has been prepared for Newmarket Gold and is based, in part, as specifically set forth below, on the review, analysis, interpretation and conclusions derived from information which has been provided or made available by Newmarket Gold, augmented by direct field examination and discussion with former employees, current employees of Newmarket Gold.

The Qualified persons have reviewed such technical information and determined it to be adequate for the purposes of this Technical Report. The Qualified Persons do not disclaim any responsibility for this information.

SRK has not performed any sampling or assaying, detailed geological mapping, excavated any trenches, drilled any holes or carried out any independent exploration work.

SRK did undertake geological and geotechnical logging of specific drillholes to assist in validating project assumptions.

4 Property Description and Location

The Maud Creek Deposit of Newmarket Gold described within this Technical Report is located within the Pine Creek region of the Northern Territory of Australia (Figure 4 1). There are other projects managed and owned by Newmarket Gold in the Northern Territory which are discussed further in Section 23, they include:

The Union Reefs Gold Project and Processing Plant (Figure 4 2), located approximately 170 km south southeast of Darwin accessible by the Stuart Highway, 18 km north northeast of the Township of Pine Creek.

The Pine Creek Gold Project;

The Burnside Gold & Base Metals Project:

The Moline Gold and Base Metals Project; and

The Yeuralba Gold and Base Metals Project.

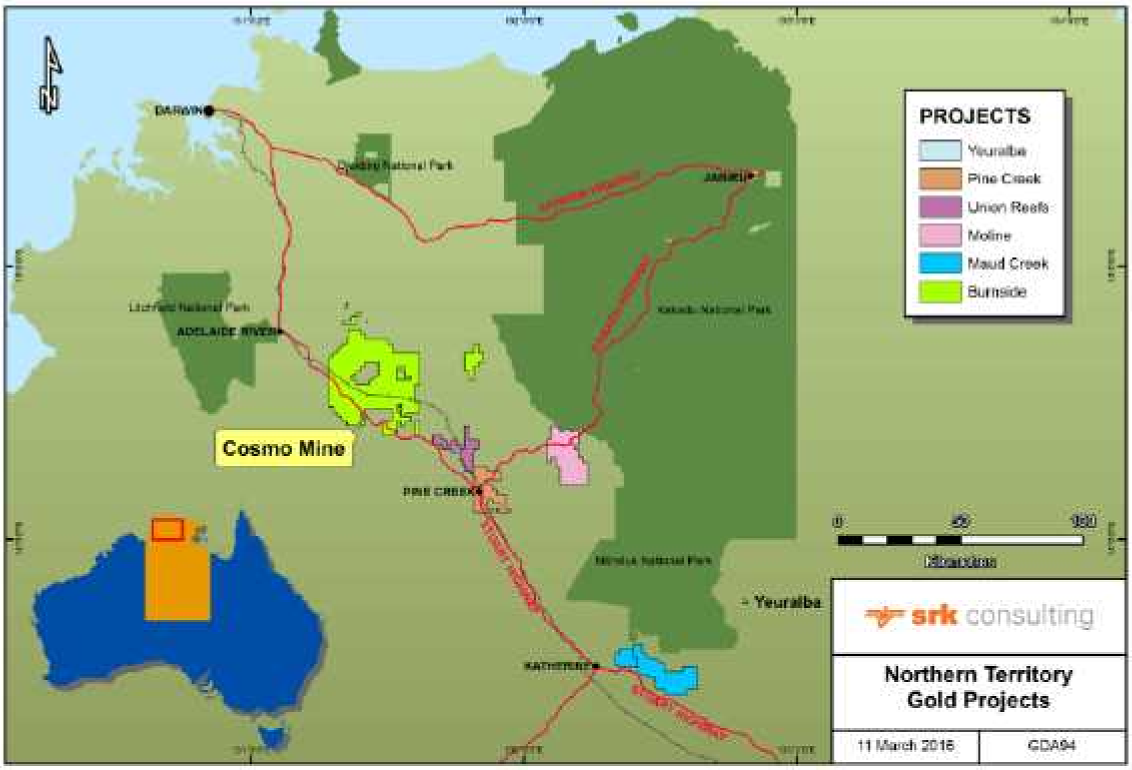

Figure 4-1: Newmarket Gold' s general location - Northern Territory gold properties

4.1 Property Location

The Maud Creek project comprises a total of 23 mineral titles (all granted) covering a total area of 29,489 ha (294.89 km2), as follows summarised in Figure 4-1.

Table 4-1: Summary of Mineral Titles Maud Creek Deposit

Licence Type	Number	Area (km2)
Exploration Licence		
Exploration Licence (EL)	2	280.26
Sub-total	2	280.26
Mineral Leases		
Mineral Lease (ML)	2	12.25
Sub-total	2	12.25
Total	**4**	**292.51**

The Project is located 20 km east of the regional administrative centre of Katherine (population 9,800) and is just east of the Township of Katherine. The deposit and proposed infrastructure is located on ML30260 which was granted 14 April 2014 and will expire on the 13 April 2024. Geographically, the Project is centred about 6.7 km straight line distance northeast of the Stuart Highway, 287 km southeast of Darwin (population 129,100), the capital city of the Northern Territory (population 233,300), at Latitude 14°26' 41"S Longitude 132°27' 10" and UTM (AMG) coordinates (WGS 84, Zone 53L) 225407mE and 8401561mS, elevation 131 m ASL.

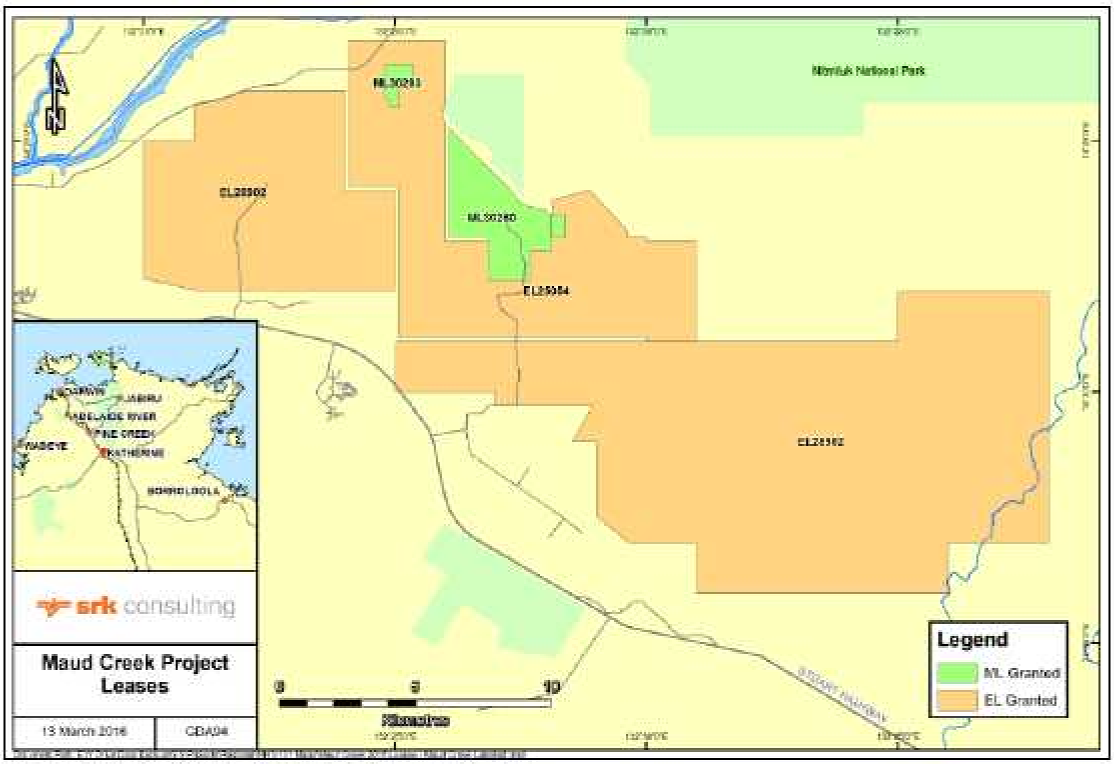

Figure 4-2: Maud Creek Gold Project Tenements

4.2 Land Tenure

The Maud Creek Project area lies within land traditionally owned by the Jawoyn people, who continue to exercise their traditional cultural attachment to the Katherine region as the owners and co-managers (with the NT Parks and Wildlife Commission) of the Nitmiluk National Park. The project lies on freehold land (NT Portion 4192, a subdivision of Portion 4159), outside lands administered by the Jawoyn Aboriginal Land Trust and the proposed mining operations area does not intersect any other Aboriginal land trust parcels.

The key mining tenement required for implementation of the Maud Creek project is ML 30260. Newmarket Gold was granted tenure over ML30260 on 14 April 2014. The current tenure expires on 13 April 2024, at which time the tenement holder has the option to renew its holdings, providing it has adhered to tenement conditions and to reporting and expenditure obligations.

There are no registered or determined native title claims over the Project area.

A land use agreement is in place for the titles shown in Figure 4-3. This is termed the Michell Compensation Agreement, which was original signed in 1992 between Michell, Biddlecombe and Trescabe to compensate the landholder for being deprived of the use of the surface of land, any damage to the property through exploration activities and being deprived of land improvements. This agreement has been assigned and accepted by Newmarket Gold.

There is an agreement between Newmarket Gold and the estate of Robert Biddlecombe relating to titles EL7775, EL8018 and MCN' s 4218 4225, (Figure 4-4), inclusive relating to a royalty payment for the mining of gold on these tenements. This is discussed further in Section 22.2.5.

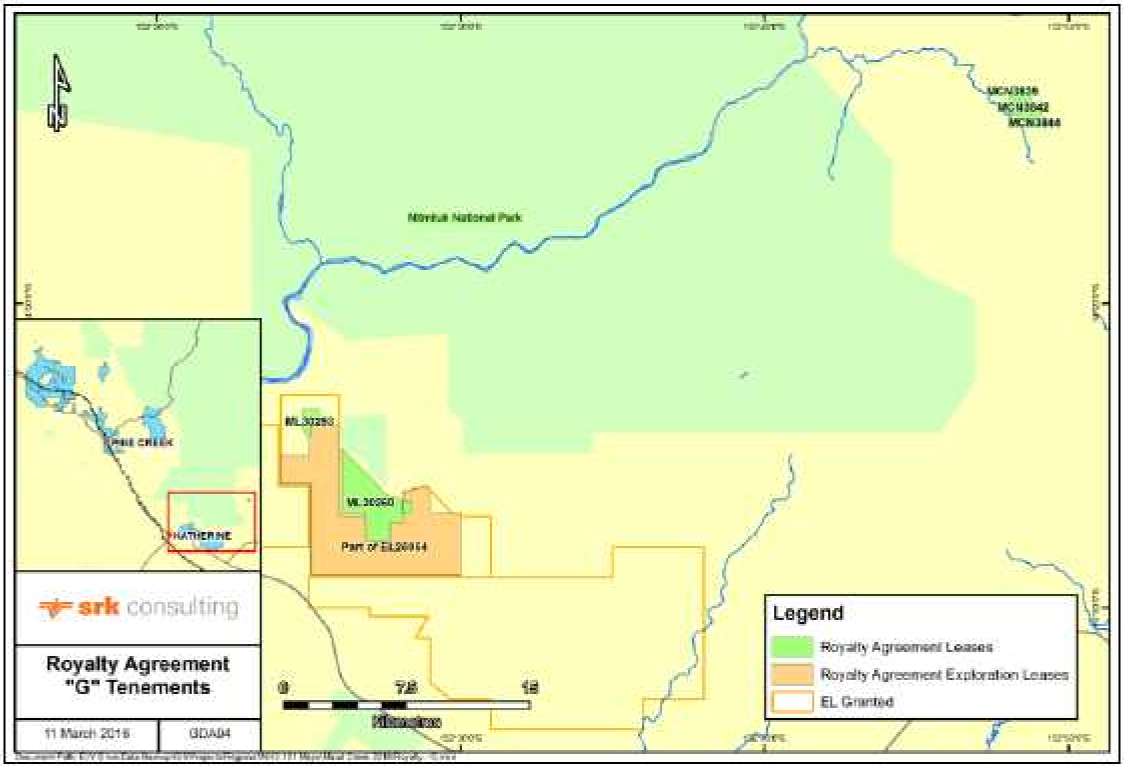

Figure 4-3: Agreements for Maud Creek Royalty

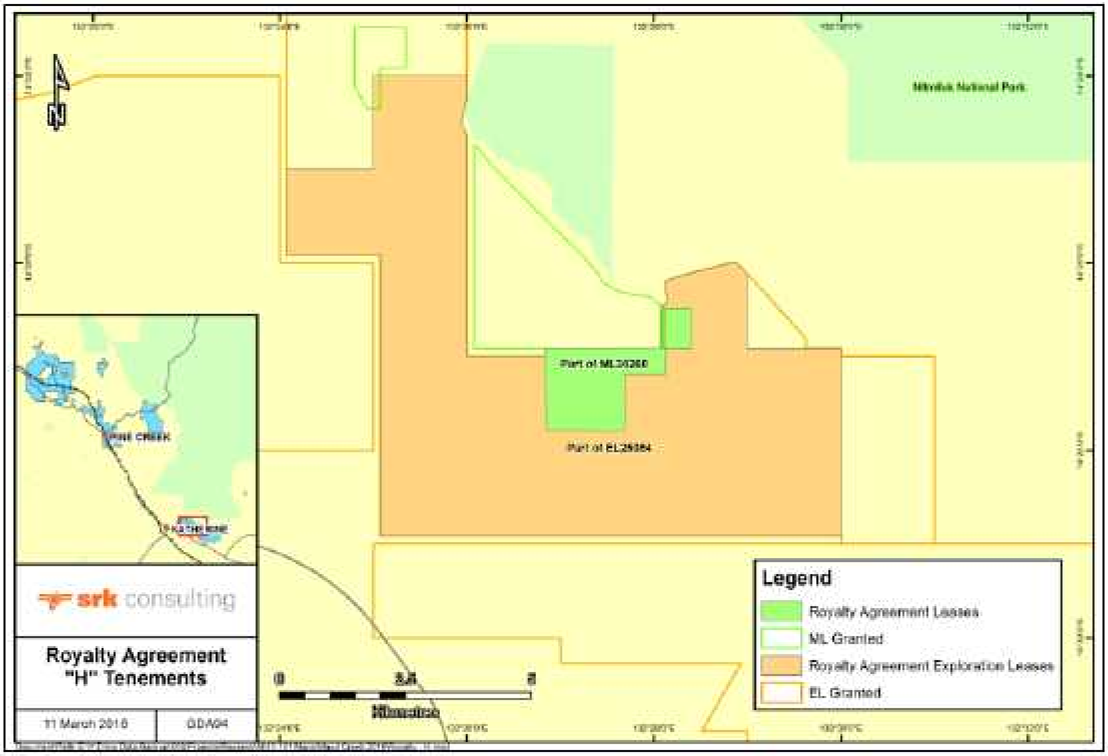

Figure 4-4: Agreements for Maud Creek Royalty with Biddlecombe

4.3 Underlying Agreements

4.3.1 Royalties

The following is a summary of the agreement and royalties, provided to SRK by Newmarket Gold, further detail is presented in Section 22.2.5.

Government Royalty - payable to the Northern Territory under the *Mineral Royalty Act (NT)*. The royalty rate is 20% of the net value of a saleable mineral commodity (in this case the gold concentrate) sold (or removed without sale) from a production unit (i.e. ML 30260) in a royalty year. The net value for the production is calculated by the following formula:

Net Value = Gross Realization - (operating costs + capital recognition deduction + eligible exploration expenditure + additional deductions)

Harmony Royalty - payable to Harmony Gold Operations Limited pursuant to a Deed of Assignment and Assumption dated 2 September 2009. Applies to all of ML 30260. The royalty rate is 1% of the value of all gold as defined in the agreement (i.e. the Perth Mint price for gold with no deductions). The royalty is not payable before 250,000 ounces of gold produced. Note also the Decision to Mine payment of AUD2M (indexed to CPI)

Virotec Royalty - payable to Mt Carrington Mines Pty Ltd pursuant to a Deed of Assignment and Assumption dated 2 September 2009. Applies only to that part of ML 30260 that was formerly within MCNs 4218 to 4225. Royalty rate is AUD5.00 per ounce with respect to 80% of the gold produced.

Biddlecombe Conglomerate Royalty - payable to Robert Biddlecombe (estate) pursuant to a Deed of Assignment and Assumption dated 2 September 2009. Applies only to that part of ML 30260 that was formerly within MCNs 4218 to 4225. Royalty rate is 1% of the gross value received as sale proceeds of all mineralization, metals, minerals and other products, after payment of the expenses incurred in smelting and refining charges.

Note that the above summaries are based on a plain reading of the documents.

NT Build Levy

Although it is not a royalty, for the purposes of the financial modelling of the Maud Creek project the construction works associated with the Maud Creek project may be subject to a levy under the *Construction Industry Long Service Leave and Benefits Act (NT)* (*Levy Act*).

In summary, the Levy Act imposes a levy on Construction Work in the Territory where the costs of construction work, commenced after 7 April 2014, are AUD1 million or more. The levy must be paid, prior to construction works commencing, by the person for whom the work is to be done (i.e. Newmarket Gold). The definition of Construction Work applies to civil works and works for buildings and structures that form part of the land, including a range of repair and maintenance works with respect to such civil works or buildings and structures.

The levy rate for construction works that commence after 7 April 2014 is 0.1% of the costs of the construction work. The Levy Act specifies that the costs of Construction Work is the total contract prices for all the construction contracts in relation to the work.

4.3.2 Farm-out Agreement

Farm-out agreements provide for third parties to explore on mineral titles, which are not owned 100% or substantially controlled by Newmarket Gold. The following discusses agreements relevant to the Maud Creek Project.

On November 6, 2013, Thundelarra Exploration Limited Uranium Exploration (Thundelarra) withdrew from a joint venture agreement with Newmarket Gold. Thundelarra was replaced by Rockland Resources Pty Ltd (Rockland) as party to the joint venture agreement, a 100% owned subsidiary of Oz Uranium Pty Ltd. Rockland was then replaced as a party to the agreement with Oz Uranium Exploration Agreement for the Pine Creek Tenements. Rockland Resources Pty Ltd (Rockland), a wholly-owned subsidiary of Oz Uranium, and Crocodile Gold formed a joint venture on November 6, 2013, in regards to uranium exploration and development on the Maud Creek, Burnside, Cosmo, Pine Creek, Union Reefs and Moline projects. Rockland has a minimum expenditure commitment of AUD1 million over the next four years. Rockland has the rights to apply for a mining tenement in its own right as long as it does not conflict with Newmarket Gold' s operations.

Over the past 24 months Rockland has been active in the Pine Creek region. They have conducted regional scale geophysical surveys and reviews (VTEM) as well as geochemical analysis, structural mapping and drilling in and around their currently identified uranium deposits. While one prospect is close to the Cosmo Mine (Fleur de Lys) no work has been conducted by Rockland on MLN993.

Land and Mining Property Swap Agreement - 2008

Land & Mining Property Exchange Deed (unregistered) and Land Use Deed dated 2008 (unregistered)

Parties involved:

GBS Gold Australia (Land Holdings) Pty Ltd and Terra Gold Mining Pty Ltd (previously Terra Gold Mining Limited)

Teelow Nominees Pty Ltd and Michael Daniel Teelow.

BACKGROUND

This agreement relates to the transfer of the Moline Project from Michael Teelow to GBS Gold in exchange for the transfer of the Maud Creek farm and NT portion 4192 from GBS to Michael Teelow. The titles transferred to GBS included MLN 41 and 1059, EL' s 23605, 22966, 22967, 22968, 22970, 24262 and 24127 and MLA 24173.

PARTICULARS

As part of the property swap agreement Teelow the lease owner has the first right of refusal on any land sale at the Maud Creek farm. The tenement owner also has the Right of way over the property which allows the tenement owner the rights to establish easement over the farm. Newmarket Gold exercised this right and has established two easements over the farm for future access to the mine.

4.3.3 Farm-in` Agreement

In 2014, Phoenix Copper Pty Ltd (now PNX Metals) entered into a Farm-in agreement with Newmarket Gold. The Heads of Agreement was signed in August 2014 and was completed in December 2014. The Farm-in agreement relates to exploration activities on the Burnside Exploration Licenses as well as at the Chessman (close to Maud Creek) and Moline projects.

The Farm-in Tenements include the Maud Creek Project including exploration licenses EL25054 and EL28902, and mineral lease ML30293.

The PNX Metals agreement does not relate to ML30260. Newmarket Gold holds 100% rights to the Maud Creek Project and PNX have no interest.

PNX Metals has been active since signing the Heads of Agreement in August 2014.

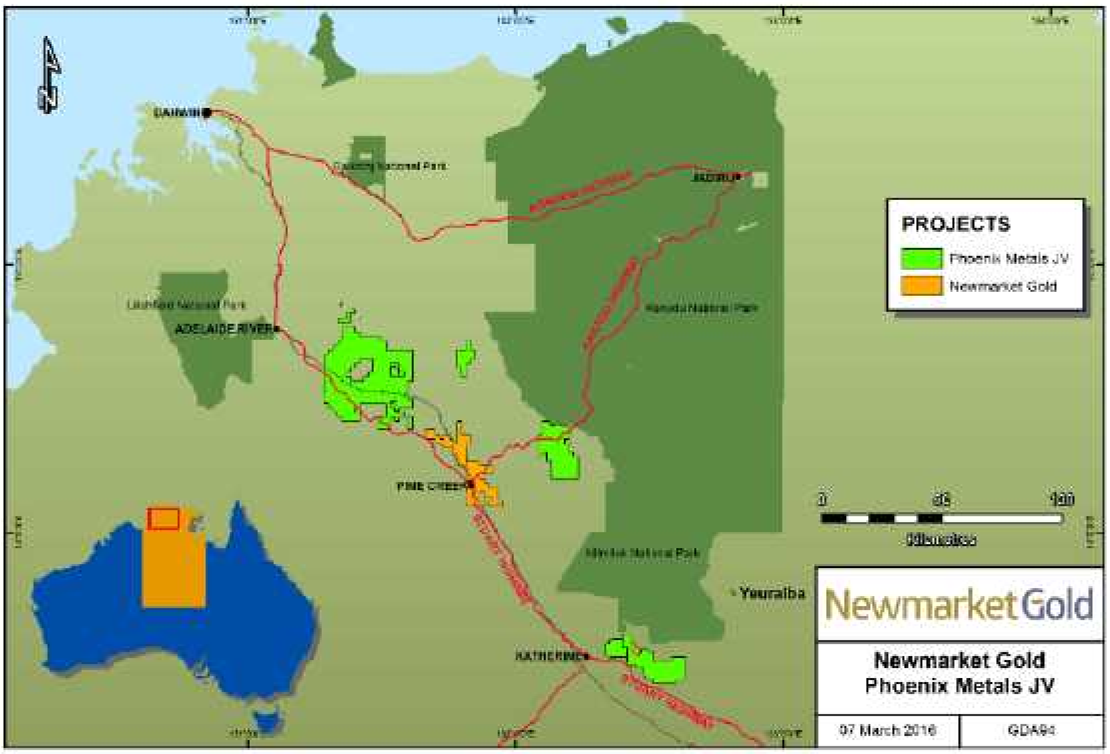

Figure 4-5: Agreements for Maud Creek Farm-in with PNX Metals

4.4 Environmental Liability

In addition to any environmental impacts that would arise in connection with new mining or mineral processing activities, Newmarket Gold would generally be liable for the management and eventual rehabilitation of legacy impacts present at the Maud Creek site at the time that Newmarket Gold took ownership of the Project.

Exploration for a range of commodities (copper, molybdenum, uranium and gold) has occurred intermittently at Maud Creek since approximately 1890. Small scale gold mining and a limited amount of processing is reported to have occurred at the site in 1890 - 1891 and again in 1932 - 1934 (Mining One, 2013).

The most recent mining at Maud Creek occurred in 2000, when Katherine Mining NL conducted open cut mining for gold. Ore was treated offsite. In the order of 9 ha of disturbed land (comprising 2.7 ha associated with the pit void, 1.6 ha associated with the former ROM pad and approximately 4.7 ha occupied by a waste rock dump) remain from previous mining. Minor disturbance related to support infrastructure (access tracks, relocatable offices) also remains (Figure 4-5).

Vegetation mapping conducted in 2007 as part of baseline environmental studies for an environmental impact assessment of the proposed Terra Gold mining project at Maud Creek mapped an area of approximately 14 ha as cleared for mining and the Terra Gold EIA reported that Approximately 96 ha of savannah woodland vegetation has been cleared in the Maud Creek project area for pastoral development and to support exploration and historical mining activity. (Crawford and Metcalfe, 2007; URS, 2008). Exploration disturbance (drillholes) arising during exploration activities subsequent to Crocodile Gold' s acquisition of the Maud Creek tenements in 2009 is reported to have been rehabilitated (Mining One, 2013).

In addition to direct clearing, historic mining activity is likely to have contributed to the establishment and spread of weeds, which are reported to be abundant and well established at Maud Creek, with dense infestations, especially along access roads and drainage lines and in disturbed areas. Some of the weeds recorded in the Project area are declared weeds under the NT *Weed Management Act*. Landholders are required to make a reasonable attempt to control and prevent the spread of declared weed species (Department of Land Resource Management, 2015).

Legacy features from previous mining at Maud Creek (notably the waste rock dump and pit void) may represent a potential source of acid or metalliferous drainage, however the limited surface and groundwater quality data for the site do not so far show the presence of significant impacts: water quality downstream of the site and in water storages at the site is generally in the range of values recorded upstream.

Figure 4-6: Existing disturbance at Maud Creek (May 2007) (from URS, 2008)

Mining activities in the Northern Territory are fully bonded. That is, the NT government requires lodgement of a security to cover 100% of the estimated cost of rehabilitating any disturbance proposed by the tenement holder under its Mining Management Plan (MMP), together with the cost of rehabilitating any pre-existing disturbance on the tenement. Additionally, from 2013, tenement holders are required to make annual contributions to the Mine Rehabilitation Fund (MRF) established under the *Mining Management Act*. The MRF payments are a non-refundable annual levy of 1% of the total calculated rehabilitation cost applied to each mining operation.

SRK has not independently estimated the cost of rehabilitating existing disturbance within the Maud Creek tenements as part of this initial review.

Permitting of the Project would require the preparation of a Mining Management Plan (MMP), including a cost estimation of mine rehabilitation and closure works. The NT Government has developed an Excel spreadsheet for estimating the security deposit to be lodged to cover mine rehabilitation and closure.

4.5 Legislation and Permitting

In the Northern Territory environmental impact assessment and subsequent authorisation and regulation of the implementation of mining and related support activities is chiefly administered until three Acts:

The Environmental Assessment Act 1982,

The Mining Management Act 2001, and

The Waste Management and Pollution Control Act 2009.

The *Mineral Titles Act* 2010 also exerts a considerable influence on regulation of environmental aspects of mining activities in that it provides for a number of significant exemptions to licensing provisions under other Acts that would otherwise apply.

If a mining proposal has the potential to give rise to significant adverse impacts on a ‘Matter of National Environmental Significance then it may also require referral to and assessment by the federal Department of the Environment (DotE) under the Environment Protection and Biodiversity Conservation Act 1999 (EPBC Act). A simplified flow chart showing the environmental impact assessment process is presented in Figure 4-6.

Although there are some differences in the duration and fine detail of the Public Environmental Review (PER) and Environmental Impact Statement (EIS) pathways, the overall processes are similar. The red arrow on the figure shows the point at which Terra Gold’ s environmental permitting was terminated by the proponent.

The Maud Creek Project currently contemplated by Newmarket Gold has not been referred to the Northern Territory Environmental Protection Authority (NTEPA) or DotE for assessment. In 2006 Terra Gold referred a proposal for mining and processing of mineralization from Maud Creek to the NTEPA for assessment. The EPA determined that Terra Gold’ s Maud Creek proposal should be assessed via the EIS pathway. A draft EIS report was issued for public review and comment in early 2008, but the proposal was ultimately withdrawn without completing the assessment process. Terra Gold also referred its Maud Creek proposal to the Commonwealth in 2006. In early 2007, the federal government determined that the Project was not a ‘controlled action’ under the EPBC Act.

It is likely that any future assessment of the Maud Creek project would follow a similar assessment path. Although much of the information produced for Terra Gold’ s EIS would still be relevant, it would almost certainly be necessary to recommence the Project’ s environmental assessment from the NOI stage (i.e., it would not be possible to re-activate the project assessment starting at the public exhibition phase). There is no fixed statutory timeline for the EIS process. A minimum of 18 to 24 months is typically required to progress from the NOI stage to completion of the EPA assessment.

Once the EIS process has been completed, permitting of operational aspects of the Project would be administered primarily under the *Mining Management Act* and the *Waste Management and Pollution Control Act.* Apart from the approvals required under these Acts, a range of authorisations or implementation conditions may arise under the following legislation:

Water Act 1992

Heritage Act 2011

Northern Territory Aboriginal Sacred Sites Act 1989

Planning Act 1999

Dangerous Good Act 1998

Transport of Dangerous Goods by Road and Rail (National Uniform Legislation) Act 2011

Territory Parks and Wildlife Conservation Act 2000

Native Title Act 1993 (Cth)

Aboriginal Land Rights (Northern Territory) Act 1976 (Cth)

Additional information on permitting of specific environmental and heritage aspects of the Project is presented in Section 20.2

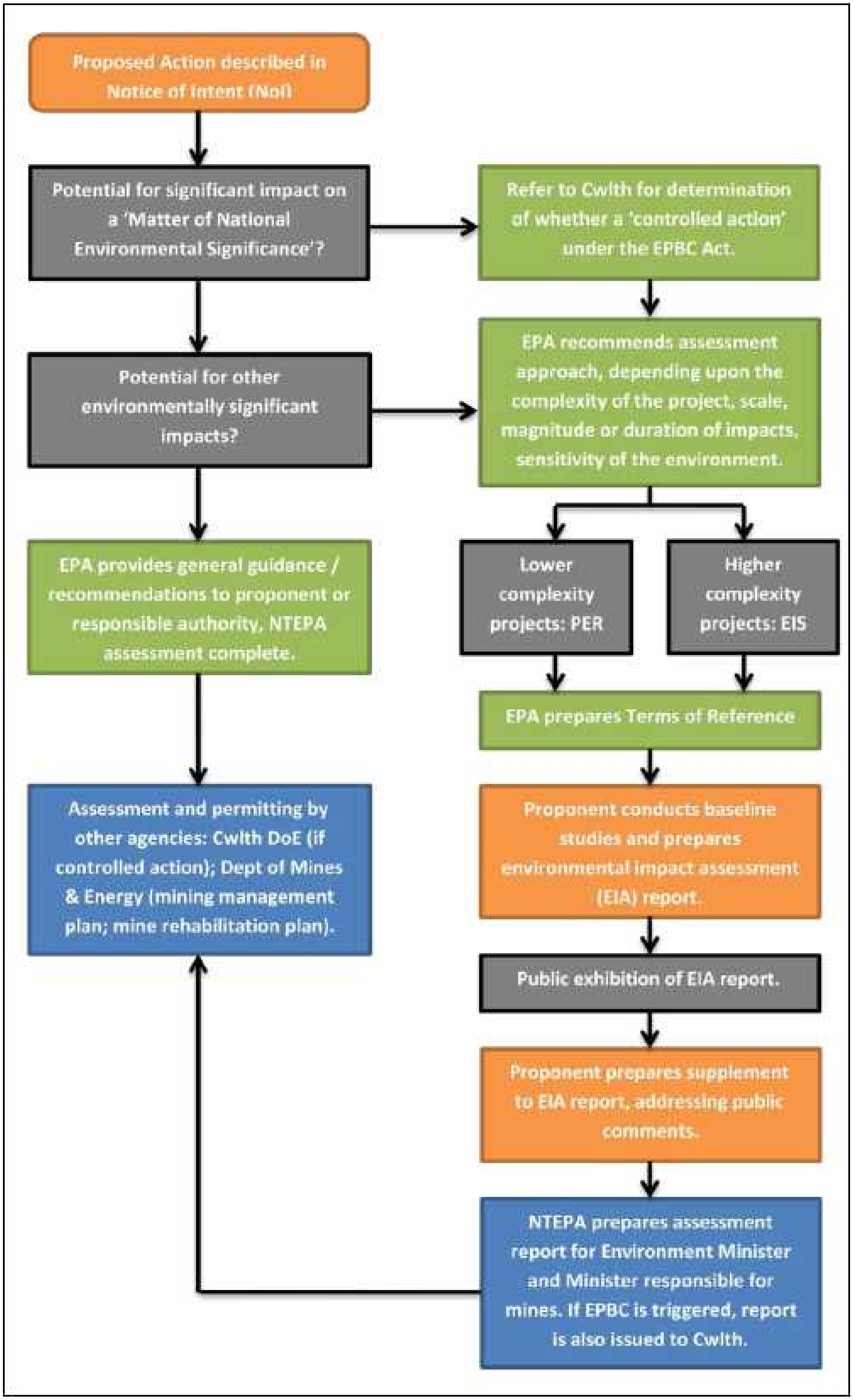

Figure 4-7: Simplified process diagram – NT environmental assessments

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Northern Territory is the least populated of all areas in Australia. It encapsulates a total area of 1.35 million square kilometers and accounts for 20% of the whole country; however, just 233,300 (ABS March 2012) or 1% of Australia's population reside there. The Territory varies considerably in topography, climate, and infrastructure. The region is dry between April and September, and wet between October and March. During the wet season everything is green and there is no dust; however, the humidity and temperatures are high and access off road is difficult. The centre is extremely arid, with greatly varying temperatures and is known as the Red Centre named because red is the predominant color found in the soil.

Darwin, Capital of the Northern Territory, lies on the coast to the north and provides the majority of infrastructure support and services for the mining industry. The Stuart Highway, which virtually bisects the country, is the main road that leads from Darwin to Alice Springs then on to Adelaide in South Australia.

5.1 Accessibility

Access is gained to the Project from Darwin by travelling south for some 314 road kilometers along the sealed Stuart Highway to the town of Katherine.

The Stuart Highway, the area' s major thoroughfare, and the Adelaide to Darwin transcontinental railway line bisect Australia in a north south sense and provide access to the Maud Creek Property. The Project site is approximately 30km from the town of Katherine.

The Union Reefs Processing Plant, owned and operated by Newmarket Gold is located approximately 185 km southeast of Darwin, 15 km north of the town of Pine Creek.

5.2 Land Use

Major land uses are traditional Indigenous uses, nature conservation (including parts of Kakadu National Park and World Heritage Area and Litchfield National Park), urban and other intensive uses and grazing. Approximately 85,000 hectares have been cleared. The region has undergone some localized clearing and the major land uses are grazing, nature conservation (including parts of Kakadu National Park and World Heritage Area and Litchfield National Park), traditional Indigenous uses and other intensive uses including horticulture.

The Daly Basin Bioregion consists of gently undulating plains and scattered low plateau remnants and has a tropical monsoonal climate with distinct wet and dry seasons and high temperatures throughout the year. Dominant vegetation is tropical eucalypt woodlands/grasslands and eucalypt open forests. Smaller patches of eucalypt woodlands and melaleuca forests and woodlands are present.

The major land use is grazing on native pastures and traditional Indigenous uses with some horticulture, grazing on modified pastures and nature conservation. The region has undergone some clearing (approximately 167,000 ha) for these developments The vegetation of the Maud Creek area consists largely of woodlands and open woodlands (predominant species – Eucalypts) that have been degraded by the impacts of cattle, buffalo and wild donkeys. No rare, threatened or endangered species have been identified in the area.

5.3 Topography

Generally the topography of the Property area is flat, locally gently undulating.

In the Maud Creek area, the terrain is flat lying to undulating. Ephemeral streams transect the project area and drain into the westward flowing Katherine River that flows all year. Land units occurring within the Maud Creek area include:

Rugged terrain with slopes 15 to 40% with shallow or skeletal soils;

Hilly terrain with slopes 5 to 15%, rocky and boulder strewn with shallow and skeletal soils;

Gently undulating crests and upper slopes to 5% with shallow rocky soils;

Undulating terrain with slopes 5 to 10% with grey and brown clays; and

Major creeks and gullied tributaries.

5.4 Climate

The Top End of the Northern Territory has a tropical monsoon climate characterized by two distinct seasonal patterns: the 'wet' monsoon and the 'dry' seasons. The wet season generally occurs from November through to April and the dry season between May and October. Almost all rainfall occurs during the wet season, mostly between December and March, and the total rainfall decreases with distance from the coast.

The mean daily maximum temperature, as recorded at Darwin on the northern coastline, is 31°C in the coolest months of June to August and 33°C in the hottest months of October and November. The mean daily minimum temperature in Darwin range from approximately 19°C (dry season) to 25°C (wet season). The average annual rainfall at Darwin is 1,713 mm.

The mean daily maximum temperature, as recorded at Katherine, is 31°C in the coolest months of June to August and 38°C in the hottest months of October and November. The mean daily minimum temperatures at Katherine range from approximately 13°C (dry season) to 24°C (wet season). The average annual rainfall at Katherine is 971 mm.

During the wet season, high intensity rainfall events are common, resulting in local flash flooding of ephemeral streams and watercourses. Mining operations are continuous throughout the year; however, increased stockpiling is undertaken in the lead up to the wet season thereby offsetting the reduced mining movements over that period. Experience has shown that it is best to shut down hauling during periods of extreme rainfall as damage to haul roads by large trucks may occur quickly.

The annual evaporation rate remains high throughout most of the Northern Territory, ranging from 2,400 mm to 4,000 mm per annum. Monthly evaporation exceeds rainfall for eight months of the year at the coast increasing to the whole year inland. It remains relatively high even during the wet season.

Climate gradually moves from seasonally wet tropical in the north to arid in the south, with corresponding changes in landscape, with areas of rocky escarpment and plateau which break a low relief in the north and rocky ridges in the south.

The Northern Territory has a diversity of vegetation that is maintained by its variety of climate and soils. Natural vegetation of the Properties is typical of savannahs of the northern part of Australia, dominated by Eucalypt species with a grassy understorey dominated by sorghum species. The Northern Territory is the only area in Australia that does not have conspicuous temperate flora.

In the north, the vegetation is typically tropical savannah (eucalypt woodland and eucalypt open woodland with a grassy understory). This landscape experiences dramatic seasonal changes with intense growth in the wet season (summer) and widespread fires in the dry season (winter). Famous worldwide for the tropical wetlands and rugged sandstone escarpments of Kakadu National Park, the wetlands are of importance for conservation, providing breeding areas, habitat and refuge for important wildlife populations.

From the north, a transition area moves from eucalypt woodlands into areas of melaleuca and acacia forests and woodlands and south into the spinifex (hummock grasslands), Mitchell grass (tussock grasslands) and acacia woodlands and shrublands. The vegetation increases in diversity around Alice Springs with areas of mulga, mallee, chenopods, hummock grasslands, small pockets of eucalypt woodlands and salt lakes.

5.5 Infrastructure and Local Resources

Darwin has a population in excess of 129,000 and is the capital city of the Northern Territory. It is the administrative centre of the Northern Territory government and a major transportation hub, with an international airport and deep water port and the Adelaide to Darwin transcontinental railway terminating at the East Arm port. As the largest city in the Northern Territory, Darwin also has excellent schools, hospitals, and retail, commercial and light industrial services.

A considerable proportion of consumer and other goods reaching the Northern Territory are brought by road from Queensland or South Australia. The Stuart, Arnhem, Kakadu, Barkley and Victoria Highways ensure high service levels to the Darwin region from the Australian capitals and other regional centres.

Despite its low population, the area between Darwin and Katherine in the Northern Territory is well serviced with infrastructure. Significant mining operations have been developed in the area over the past 30 years, with gold mining and processing operations conducted within or in close proximity to the project areas at Cosmo Howley, Brocks Creek, Pine Creek, Mount Todd and Union Reefs. Katherine is a regional centre with a population of approximately 9,800 and enjoys excellent infrastructure, services and communications. This is the closest centre of population to the Maud Creek project.

The regional mining communities of Pine Creek (with a population of 450) and Adelaide River (population of 200) support the Burnside, Maud Creek and Moline gold projects.

The Arnhem Highway to the east southeast of Darwin provides a communication link to the Kakadu National Park and Jabiru, a town of 1,135, which provides accommodation for the uranium mines in the vicinity. Accommodation and services are available along the highway, primarily for the tourist trade.

6 History

6.1 Introduction

Gold was initially discovered in the Maud Creek area in 1890 and a small plant was set up but ultimately abandoned in 1891. This is now called the Chlorite Hills and O' Shea' s area.

The area was re looked at from 1932 34 when 400 tonnes of ore produced 540 ounces of gold. Mining was from about 20 shallow shafts and small holes that were 6 12 m deep with horizontal workings from 15 30 m in length in the Chlorite Hills and O' Shea' s area.

Interest in the Maud Creek area was rekindled in the 1960s during an assessment of the mineral potential of the Top End of the Northern Territory. This study was prompted by the discovery of significant uranium mineralization in the nearby South Alligator River valley in the mid 1950s.

The Northern Territory Geological Survey carried out IP surveys, soil sampling and petrographic investigations in the late 1970s as part of an assessment of an extension to the nearby township of Katherine.

6.2 Ownership and Exploration Work

Between 1966 and 1973 several companies including Western Nuclear Australia and Magnum Exploration NL explored the area for copper, gold and uranium. IP surveys and drilling of siliceous and gossanous breccias intersected low, albeit anomalous, concentrations of copper and molybdenum and numerous pyritic zones.

In 1973 Magnum Exploration NL (EL147) explored the breccia in the Red Queen/Chessmen area as part of a copper uranium search. They considered the breccia to be similar to the Rum Jungle occurrence. They drilled 7 holes into the breccia and met with pyritic material with low copper values. They also dug trenches, and obtained anomalous copper and molybdenum values. They did not assay for gold.

In 1985 the Minerals and Exploration and Development Group (MEDG) of CSR Ltd explored the Maud Creek area. Stream sediment sampling returned a 1.3 ppb BLEG gold result about 1.3 km to the west of the old 19th century workings on Maud Creek (now called Chlorite Hills and O' Shea' s). Placer Exploration purchased MEDG in 1987 and followed up on the BLEG anomaly with rock chip sampling and drilling which subsequently resulted in the discovery of the Gold Creek Zone.

Placer sold the deposit in 1992 to Kalmet Resources NL.

CSR Limited in 1986 held the Peckham Hill EL4874 that covered the Chessmen/ Red Queen prospect (located to the northwest of the Gold Creek Zone). They recognized that the breccia in the area previously mapped by the Bureau of Mines displayed epithermal textures. They conducted exploration programs comprising rock chip sampling, soil sampling, trenching and drilling at Red Queen and along strike. Some 10 km of strike of the breccia/veins were rock chipped and anomalous gold in a gossan sample was reported at Red Queen up to 6.63 g/t Au and 1.02% arsenic (AMG 8406034mN 221372E). The samples were also anomalous in copper, antimony, mercury and thallium.

Another gold anomalous breccia was located 1 km NE of Red Queen and this assayed 0.23 g/t Au. The remainder of the siliceous breccia gave values below 0.07 g/t Au. The north Chessmen location gave an anomalous stream sediment value (14.3 ppb gold at AMG 8406800N 331300E).

Soil sampling over the Red Queen prospect produced a peak value of 0.33 g/t Au at the western edge of the volcanic sediment contact. The anomalous breccia also gave positive soil values. Another zone was detected in the north with a value of 0.32 g/t Au in sediments, but without obvious structural association.

Trenching by CSR was carried out in the Red Queen area to test the soil anomalies numbered T1 to T7. T1 coincided with a Magnum Exploration (1973) trench and anomalous gold, arsenic and copper were reported with the best gold value at 6.72 g/t Au from quartz veins in mafic fragmentals. T2, also in an adjacent Magnum trench gave a peak value of 0.61 g/t Au. Trench T3, located 30 m to the SW, met with a carbonated zone with a maximum value of 1 m @ 0.42 g/t Au. T4, on a soil anomaly, met with 2 m @ 0.33 g/t Au. T5 and T6 did not explain the soil anomaly. T7 met with a best result of 2 m @ 0.46 g/t Au.

Percussion drilling was carried out totalling 1,210 m in 9 holes (CMPDH series). While promising mafic lithologies and chalcedony quartz carbonate alteration were met with, the results were sub economic with the best values falling in the range 0.1 to 0.4 g/t Au over intervals up to 19 m.

Between 1989 1990 Placer re established the CSR grid at Red Queen/Chessmen and conducted soil sampling (412 samples) on a 25 m grid. Samples were assayed for gold, copper, lead, zinc and arsenic. They reported two zones of gold anomalism, one over quartz veins and sheared chert and the other with the western contact zone. The anomalies displayed little correlation with the CSR soil anomalies. Elevated base metal values appeared to correlate with the hematite stained volcanics.

Rock chip sampling and mapping comprised 29 samples and assays up to 4.1 g/t Au were obtained from hematite float in the western contact zone.

Two lines of IP, dipole dipole, were carried out. A chargeability anomaly was attributed to black cherts. Ground magnetics were carried out over two airborne magnetic anomalies.

Five RC holes were drilled at Red Queen for 576 m. (RQP) Hole RQP5 drilled under a soil anomaly near the IP chargeability anomaly met with 10 m @ 0.95 g/t Au from 46 m and 8 m @ 0.97 g/t Au from 60 m. The associated lithology was black cherts. In general, attempts to correlate surface geology with the drilling proved confusing.

The Chessmen/Red Queen area was reduced to three MCNs and held as a second priority resource area following the discovery of the Main Zone gold deposit at Gold Creek. The trenches and drillholes were rehabilitated.

The Maud Creek project was owned by a number of companies until the acquisition of the project by GBS Gold in December 2006. Substantial drilling, in the order of 66,000 90,000 m of RC and diamond drilling, is reported on the project area during the period 1966 - 2006, oriented toward gold exploration.

During 1985 and 1986, CSR Limited (CSR) explored the area in an attempt to locate gold mineralization in the Lower Proterozoic dolerites. Work included 25 metre line spacing airborne magnetic and radiometric surveys, stream sediment sampling, soil sampling, rock chip sampling, petrographic sampling and trenching.

Between 1993 and 1997, Kalmet Resources NL (Kalmet) completed a series of drilling programs at Maud Creek. Metallurgical testing of five high grade RC samples was completed and an environmental impact study was commissioned. Metallurgical studies showed the primary gold bearing sulphide mineralization was refractory in nature and bio oxidation tests were initiated. A close line spaced airborne magnetic and radiometric survey was contracted over a significant part of the land position (Figure 6-1). Interpretation of this radiometric data was completed by Independent Engineers in 2005 (Figure 6-2 and Figure 6-3).

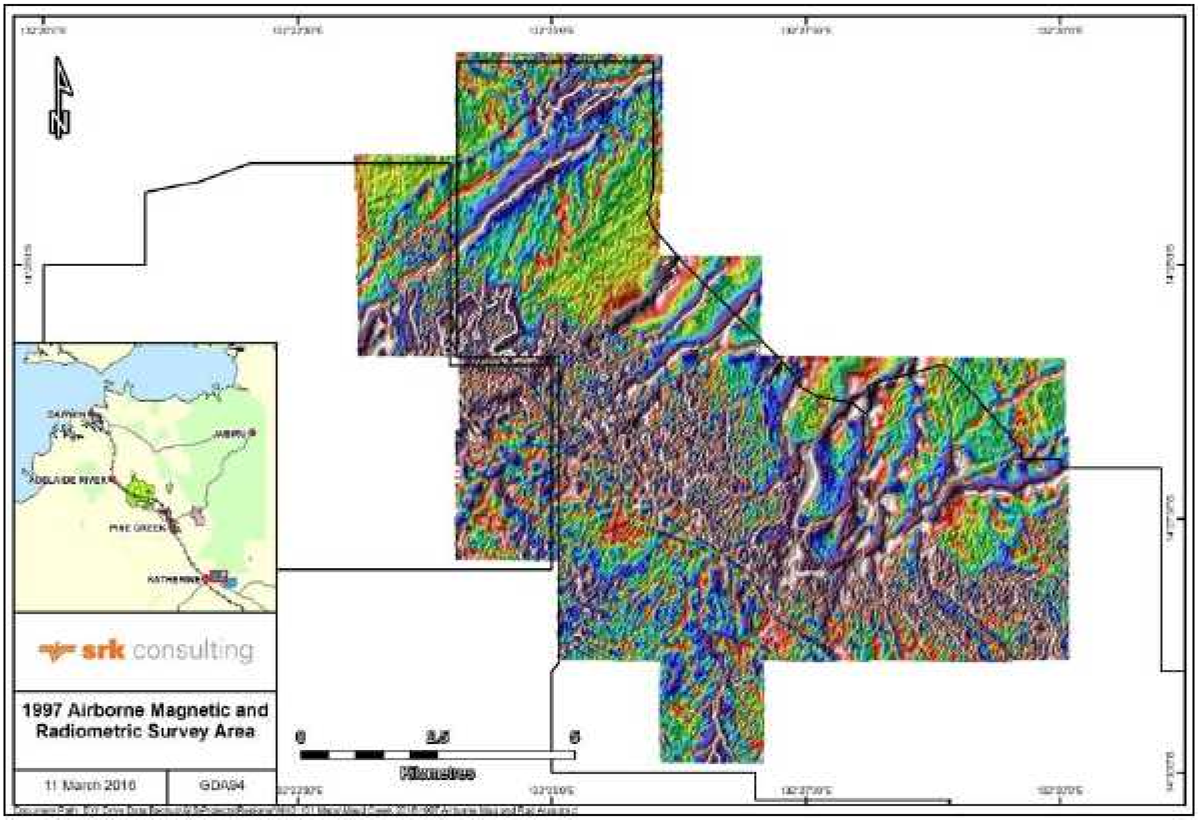

Figure 6-1: 1997 airborne magnetic and radiometric survey

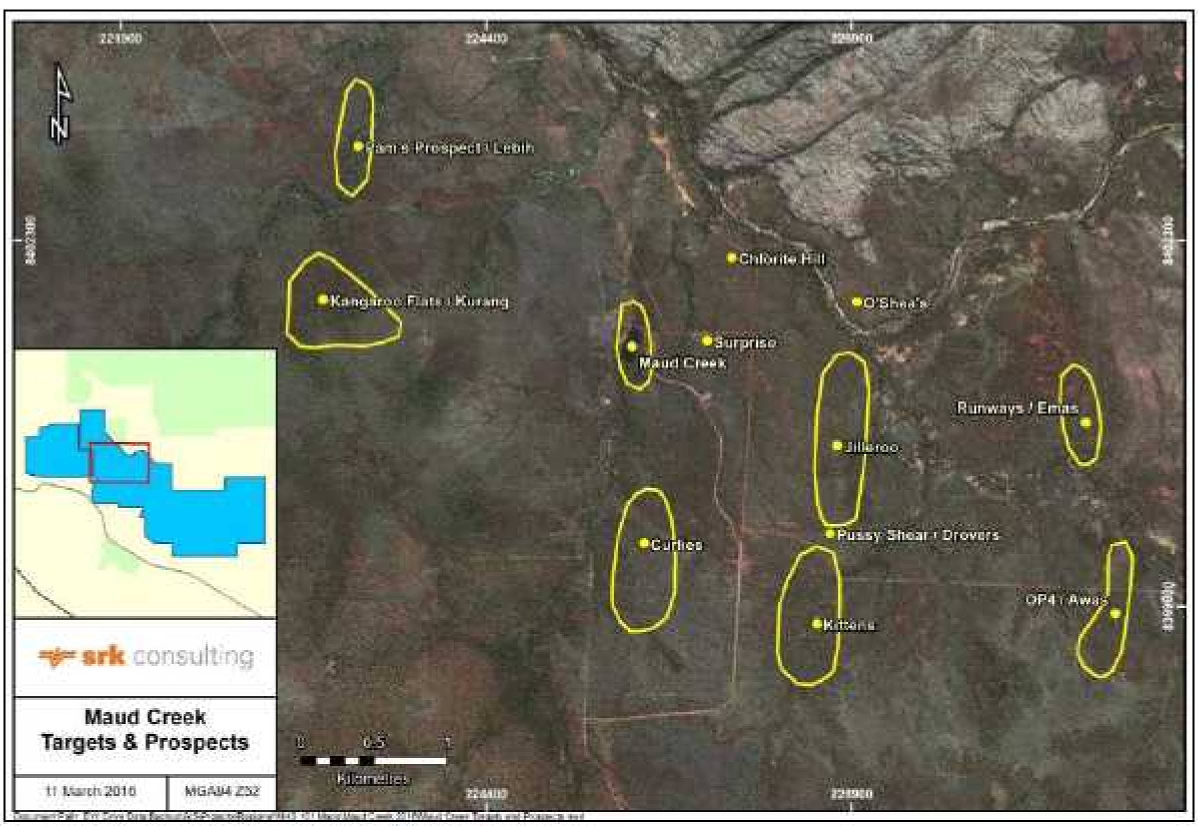

Figure 6-2: Maud Creek area pits and prospects (Independent Engineers)
Note: interpreted from radiometric data

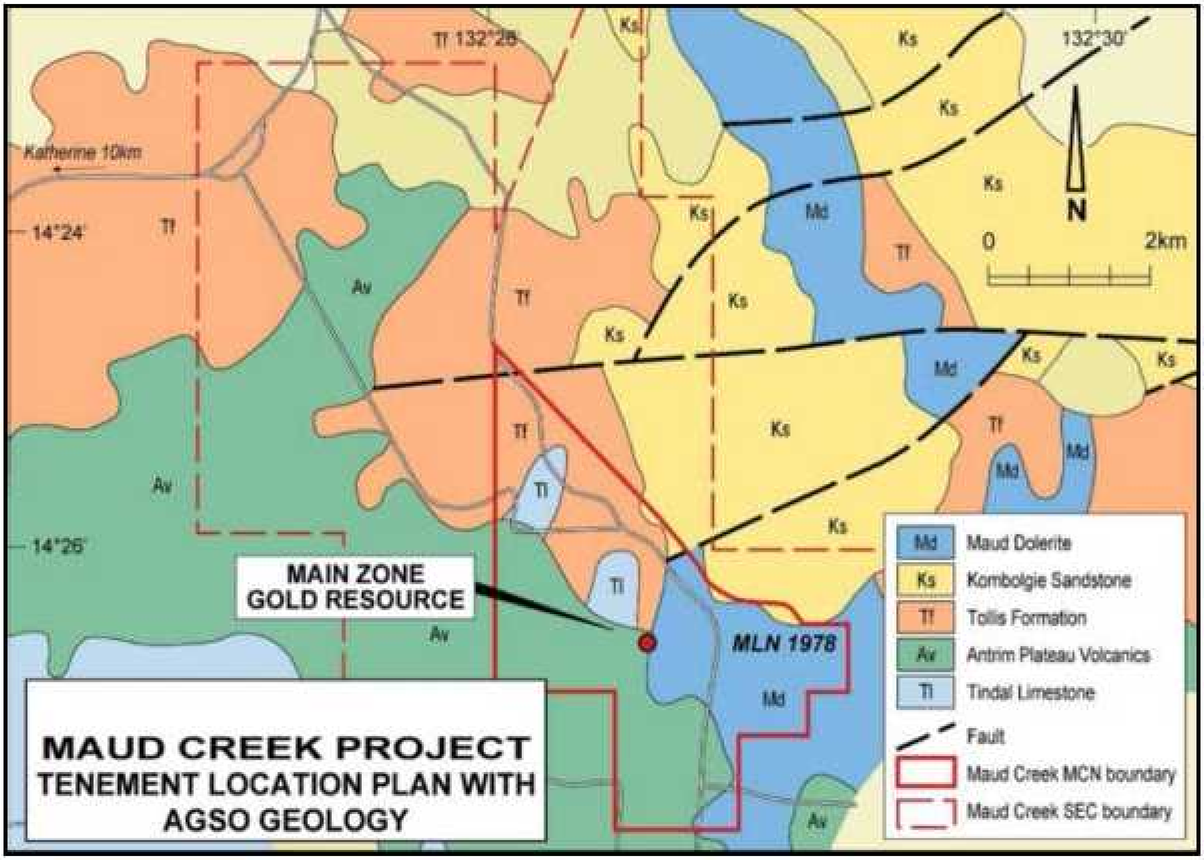

Figure 6-3: Maud Creek regional geology and structural interpretation by Independent Engineers 2005

In 1997, Kilkenny Gold NL acquired Kalmet and undertook RC and diamond drilling. They carried out significant drilling and increased the global resource to 995,000 ounces (Indicated and Inferred Mineral Resources).

Further metallurgical test work was completed including pilot scale flotation and bio oxidation program. In 1998, Signet Engineering completed a full feasibility study for the extraction and processing of oxide, transition and primary mineralization from Maud Creek. A comprehensive draft Environmental Impact Study was also produced.

In 1998 Kilkenny Gold commissioned SRK to complete a structural assessment and interpretation of aeromagnetic results. The report, maps and interpretation are quite detailed.

A major flood in the Katherine area in 1998 saw the loss of a significant amount of technical data in the form of reports and diagrams.

AngloGold acquired rights to mine the oxide zone of the Main Zone deposit at Maud Creek and treat the ore at the Union Reefs plant. Mining operations were conducted during 2000. A total of 173,581 tonnes grading 3.32 g/t Au for 18,527 ounces were obtained. Ore was trucked from Maud Creek to the Union Reefs mill.

Hill 50 Gold NL acquired the Maud Creek project from Phoenix Mining Ltd in March 2001 and conducted an extensive review of previous exploration, which identified five gold targets within the property. A program of rock chip sampling was conducted at the Runways prospect. Additional RC and diamond drilling was completed at Gold Creek and surrounding prospects.

In late 2001, Harmony Gold Company Ltd launched a takeover bid for Hill 50 and by mid 2002 had successfully completed the acquisition of the company and all its assets including the Maud Creek deposit. A photo geological interpretation of the property was completed by Snodin (2002), who recommend areas for further follow up work.

In December, 2004, and prior to being acquired by GBS Australia, Terra Gold Mining Pty Ltd (TGM) purchased an option to acquire the Maud Creek project from Hill 50, which by that time had been acquired by Harmony Gold. In January 2005, TGM drilled a single combined percussion diamond hole into the mineralized Main Zone to supply a limited quantity of sample for metallurgical test work purposes. Following a preliminary due diligence examination, in May 2005 TGM exercised its option to purchase the Maud Creek project.

In 2005, four holes totalling 711 m consisting of 406 m of RC pre collar and 305 m of HQ3 diamond drill core were completed. These holes were designed primarily to provide additional samples for metallurgical test work.

In August 2005, GBS Gold Australia Pty Ltd, a wholly owned subsidiary of GBS Gold, announced its intention to acquire all of the issued share capital of Terra Gold Mining, including its interest in the Maud Creek Gold Project. The acquisition was completed in January 2006 and the 100% interest was transferred to GBS Gold.

GBS completed resource calculations as well as mining, geotechnical and hydrogeological studies of the Maud Creek deposit. They also completed an extensive EIS report on the deposit area.

An agreement to acquire a number of properties, including the Maud Creek property, was entered into on June 19, 2009 from GBS Gold International Inc. (GBS Gold) (In liquidation). GBS Gold operated the Tom' s Gully and Brock' s Creek underground gold mines, mined several open pit gold deposits and operated two gold Processing Plants, one at Tom' s Gully, the other at Union Reefs, near Pine Creek, Northern Territory, until September, 2008, when administrators were appointed.

On November 6, 2009, the mining tenements including the Maud Creek Property were registered in the name of Crocodile Gold, which became Newmarket Gold in July 2015.

7 Geological Setting and Mineralization

7.1 Regional Geology

The Maud Creek Project lies within the Archean to Paleoproterozoic Pine Creek Orogen (PCO) which is located in the north of the Northern Territory and extends from Katherine in the south to Darwin in the north (Figure 7-1). The PCO is exposed over 47,500km^2 and consists of a deformed and metamorphosed sedimentary basin with a thickness of over 4 km and overlies a Neoarchean (ca 2670-2500 Ma) granitic and gneissic basement (Ahmad & Hollis, 2013). The PCO hosts over a thousand mineral occurrences and is recognised as one of the most prospective mineral provinces within Australia (Ahmad & Hollis, 2013). Known resources include uranium, gold, and platinum group metals (PGMs), as well as substantial base metals, silver, iron and tin-tantalum mineralization.

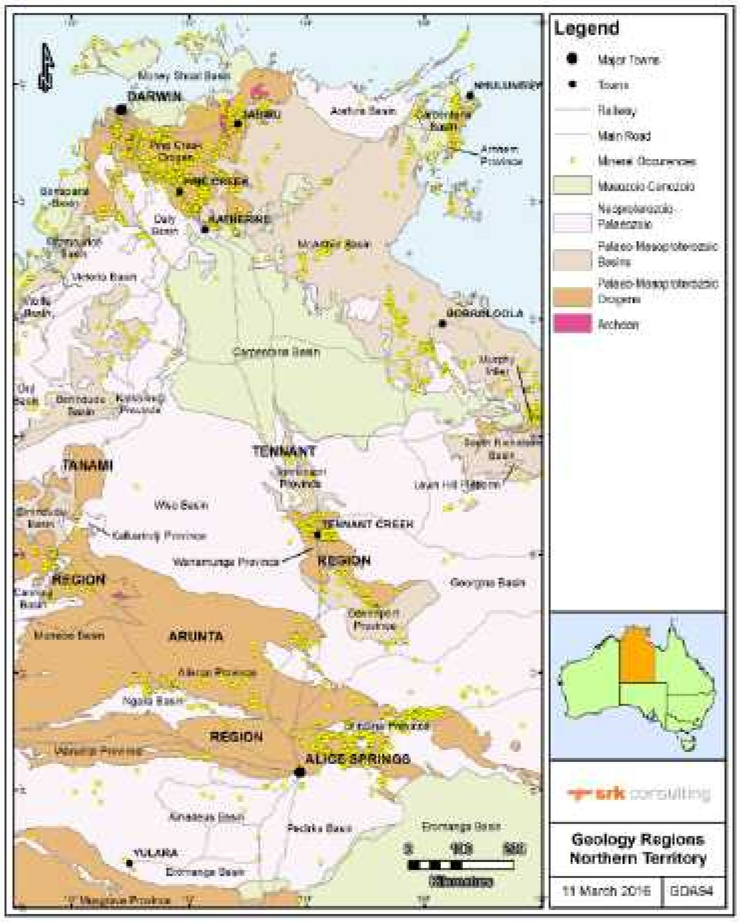

Figure 7-1: Pine Creek Orogen (northern orange zone) within the Northern Territory

Source: Ahmad & Hollis, 2013

The basement terrain of the PCO consists of a series of late Archean granite-gneiss basement domes which have subsequently been overlain by fluvial to marine sedimentary sequences of the Paleoproterozoic. These sequences have been divided into the Woodcutters and Cosmo Supergroups, which are separated by a major unconformity representing a time break of 160 Ma (Ahmad and McCready, 2001). Several highly reactive rock units are included within Cosmo Supergroup including carbonaceous shale, iron stones, evaporite, carbonate and mafic to felsic volcanic units of the South Alligator and Finniss River Groups. A northwest trending fabric is evident throughout these sequences resulting from greenschist facies metamorphism and multiphase deformation. A period of widespread felsic volcanism in aureoles between 500 m and 2 km wide overprint the earlier regional metamorphism following deposition of the Cosmos Supergroup (Snowden Report, 2008). A period of extension deformation following intrusions of these granitoids resulted in an extensive array of northeast and northwest trending dolerite dykes intruding the metasedimentary sequences.

Gold mineralization within the PCO is defined as orogenic in nature and are recognised to have common geological, geochemical, mineralogical and thermochemical characteristics (Ahmad & Hollis, 2013). Gold mineralization is commonly strongly structurally controlled within the region with gold exploiting structures such as anticlines, strike slip shear zones and duplex thrusts as well as located in proximity to the Cullen Granite Batholith (Snowden Report, 2008). Mineralization is commonly recognised within the upper Woodcutters Supergroup and Cosmos Supergroup, specifically within the South Alligator Group and lower parts of the Finniss River Group. Of particular stratigraphic importance for mineralization are the Wildman Siltstone of the mount Partridge Group and the Koolpin Formation, Gerowie Tuff, and Mount Bonnie Formation of the South Alligator Group and the Burrell Creek Formation of the Finniss River Group as well as the Tollis Formation (Figure 22) (Snowden Report, 2008). Goldfields hosted within these units include Pine Creek, Mount Todd, Howley, Golden Dyke, Maud Creek and Brocks Creek gold fields (Ahmad & Hollis, 2013). Descriptions of these prospective host units are briefly summarised as follows.

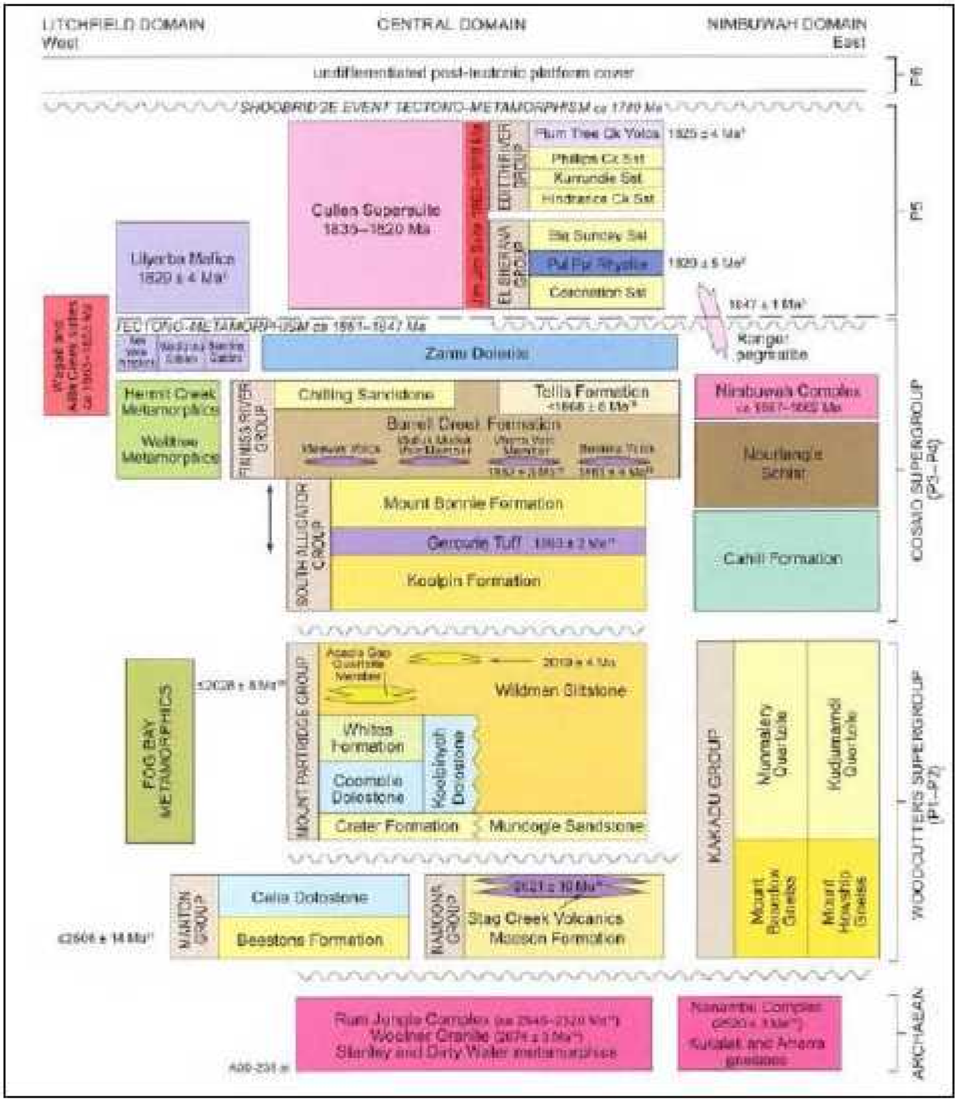

Figure 7-2: Summary Stratigraphic Chart of the Pine Creek Orogen

Source: Ahmad & Hollis, 2013

The Wildman Siltstone is the upper most unit of the Mount Partridge Group of the Woodcutters Supergroup and consists of a succession of laminated banded silty pyritic carbonaceous phyllite with minor sandstone and tuff beds, with an overall thickness of approximately 1,000 m. This unit is unconformably overlain by the Koolpin Formation of the South Alligator Group (Ahmad & Hollis, 2013).

The South Alligator Group is the oldest member of the Cosmos Supergroup and is divided into three units (Koolpin Formation, Gerowie Tuff, and Mount Bonnie Formation). Compositionally this group consists of a succession of iron rich sedimentary rocks, tuff, carbonate rocks, shale, greywacke and siltstone (Snowden Report, 2008). The Koolpin Formation is the lowermost unit of the South Alligator Group. It consists of sulphidic and carbonaceous argillite, ferruginous chert, ironstone, silicified dolomites and phyllitic mudstones which were deposited in a low energy environment (Ahmad & Hollis, 2013). The Koolpin Formation varies in thickness from less than 300 m to in excess of 1000 m. The Gerowie Tuff is up to 750 m thick and is comprised of mudstone, siliceous shale, siltstone and tuff with subordinate amounts of laminated cherts and carbonaceous siltstones. Minor quartz nodules and iron rich sedimentary sequences are additionally recognised (Ahmad & Hollis, 2013).

Numerous semi-conformable sills of pre-orogenic Zamu Dolerite intrude the Koolpin Formation and the Gerowie Tuff and vary in thickness from several meters to a few hundred meters. The Mount Bonnie Formation is the uppermost unit of the South Alligator group and consists of greywacke, carbonaceous siltstone, chert, tuff and ironstone and with a variable thickness between 150 and 400 m thick (Ahmad & Hollis, 2013).

The Burrell Creek Formation is the lowermost sequence within the Finniss River Group and is comprised of a thick (<3000 m) sequence of turbiditic sediments including greywackes, siltstones and mudstones (Snowden Report).

The Tollis Formation is the youngest member of the Finniss River Group and is host to several gold deposits including Maud Creek, Mount Todd, and Quigleys deposits (north, south, extended) (Ahmad & Hollis, 2013). This unit has limited aerial extent and consists of a succession of interbedded mudstone, slate, metagreywacke and minor felsic volcaniclastic shale that was conformably overlies the Burrell Creek Formation. This unit has previously been attributed to the El Sherana Group, however more recent interpretations of have placed this unit within the Finniss River Group (Ahmad & Hollis, 2013).

7.2 Property Geology

The Maud Creek gold field lies approximately 20km to the east of Katherine and lies within the south-eastern part of the Pine Creek Geosyncline (Figure 7-3). The Maud Creek goldfield hosts the historic Maud Creek Mine and the Maud Creek deposit (historically known as the Gold Creek deposit) (AngloGold Report, 2000). Proterozoic rock units in the Maud Creek area comprise the Tollis Formation, Maud Dolerite, Dorothy Volcanics (formerly Dorothy Volcanic Member), Edith River Volcanics, and Kombolgie Formation.

The Maud Creek deposit is hosted within the Tollis Formation which outcrops in the centre of the northwest of the Maud Creek area. This unit is typified by thin to thick beds of alternating greywacke and mudstone, minor conglomerate, altered mafic to intermediate volcanic rocks and banded ironstone. The Dorothy Volcanics consist of basaltic lava, pyroclastic rocks, tuffaceous sediments and sills and locally lie in faulted contact with the Tollis Formation (Ahmad & Hollis, 2013). To the east of the Maud Creek Deposit, the Maud Dolerite intrudes the Tollis Formation and outcrops as irregular bodies of up to 200m in width (Snowden Report, 2008).

In the northern portion of the Maud Creek area, felsic volcanics of the Edith River Group are unconformably overlain by the fluvial sediments of the Kombolgie Formation. In the south and west the Tollis Formation is masked by Cambrian Antrim Plateau Volcanics and Tindall Limestone.

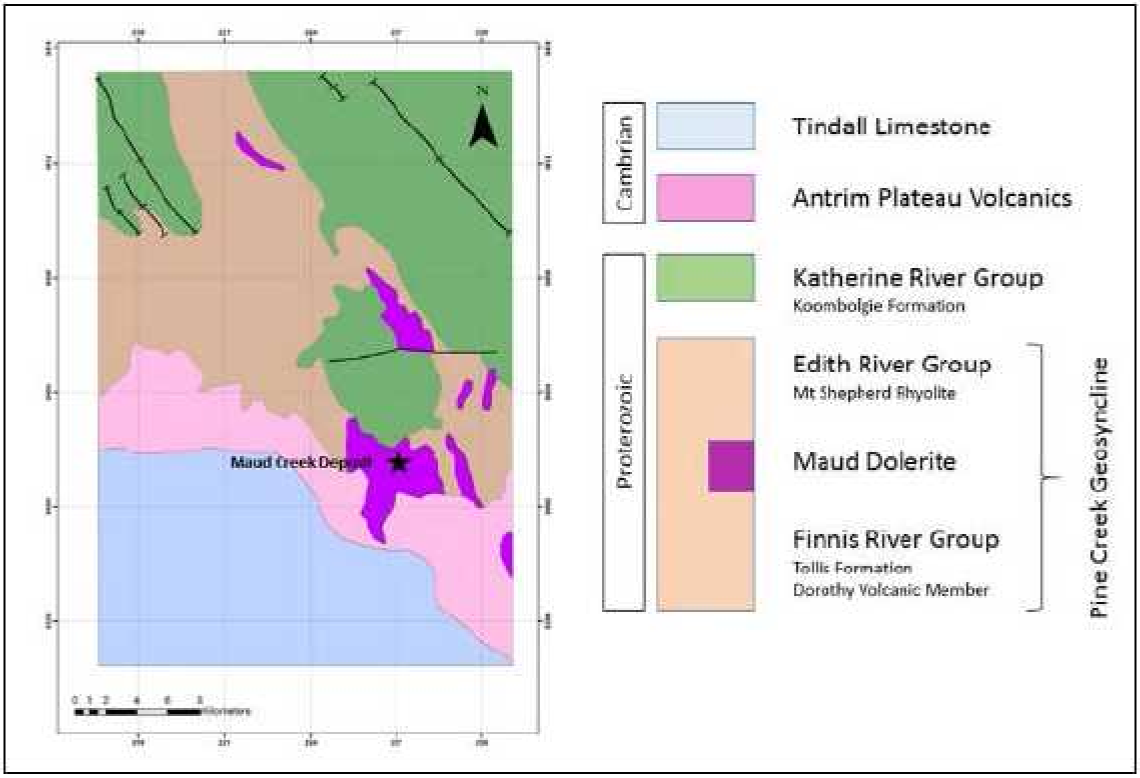

Figure 7-3: Location Map of the Maud Creek Deposit

7.2.1 Property Mineralization

The Maud Creek deposit is hosted within the Tollis Formation of the Finniss River Group. Mineralization is associated with a north-south trending Gold Creek Fault Zone (GCFZ) that forms the contact between mafic tuffs of the Dorothy Volcanics to the east and sedimentary rocks of the Tollis Formation to the west. The GCFZ and primary Maud Creek mineralization dips steeply to the east (65-75). The GCFZ is characterized by intense deformed and brecciated to catallactic zone up to 10 to 15m width (AngloGold Report, 2000). The GCFZ and Maud Creek mineralization is associated with stockworks and massive quartz veining, silica flooding and brecciation as well as intense graphitic and chloritic alteration (Ahmad & Hollis, 2013). Additional alteration recognised includes silica, carbonate, fuchsite and haematite. The contact zone deposit geometry has been defined as lenticular in shape with a steep plunge (70-80) to the south-east. This principal mineralized zone extends approximately 250 m north-south, and ranges in width from several meters to up to 50m width. The deposit remains open at depth (Snowden Report, 2008).

Mineralization is recognised to extend beyond the contact vein lodes, with dispersion up to 25m into the hanging wall tuff and 5m into the footwall sediments. Outside of the primary vein deposit minor hanging wall micro breccia zones are recognised predominantly occurring proximal to minor faulting parallel to the GCFZ (AngloGold Report, 2000).

Away from the main contact fault zone, gold is recognised within a sub-vertical shear zone which lies proximal to the contact of the Maud Dolerite (Snowden Report, 2008). Mineralization is less continuous in this zone, with the absence of any major vein lode systems evident.

Gold occurs within the deposit as both free gold and as refractory gold in pyrite and arsenopyrite (Snowden Report, 2008). Sulphides can constitute up to 5% of the deposit with pyrite and arsenopyrite and gersdorffite (NiAsS) recognised. These sulphides form as disseminations as well as massive intervals containing up to 50% pyrite (Ahmad & Hollis, 2013). Quartz makes up the remaining gangue mineral of the deposit assemblage.

Ahmad & Hollis, 2013, Pine Creek Orogen: Ahmed M and Munsen TJ (compilers). Geology and Mineral Resources of the Northern Territory, Northern Territory Geological Survey, Special Publication 5.

7.3 Deposit Mineralization

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8 Deposit Types

The contribution from gold deposits in Proterozoic sedimentary basins to total gold production has increased markedly over the past two decades, both globally and within Proterozoic basins in Australia. Consequently, many Proterozoic basins are now considered high priority exploration targets.

8.1 Deposit Models

A variety of genetic models, ranging from magmatic through hydrothermal to syngenetic, have been postulated in the past for the formation of gold deposits in the Pine Creek Geosyncline (Figure 8-1). Gold and base metal mineralization in the Pine Creek Geosyncline is commonly associated with granite intrusions and have often been classified as high temperature contact aureole deposits. A secondary host rock control has also been suggested due to the association of gold mineralization with carbonaceous metasedimentary rocks.

However, much of the gold mineralization occurred after the main intrusive event, the intrusion of the Cullen Batholith, and the relationship of gold mineralization and carbonaceous rocks is not the most important control on mineralization. More recently, authors have argued that gold mineralization is structurally controlled; occurring in brittle ductile structures at the greenschist amphibole facies boundary and hence has an epigenetic origin (Partington & McNaughton, 1997).

In places, e.g. The Cosmo Howley area, duplex thrust folds with buckle folding or basin and dome structures appear to be more significantly mineralized. The presence of shear systems linking anticlines higher in the sequence also appears to have provided the ideal fluid focusing mechanisms to localize gold bearing fluids.

Accepting that gold deposits of the Northern Territory have a structurally controlled mesothermal setting, then on the basis of host rock and mineral association they can be divided into seven types:

Gold quartz veins, lodes, sheeted veins, stockworks, saddle reefs (Pine Creek Orogen)

Gold ironstone bodies (Tennant Inlier)

Gold in iron rich sediments (Pine Creek Orogen, Tanami)

Polymetallic deposits (Iron Blow, Mt Bonnie)

Gold PGE deposits (South Alligator River area)

Uranium gold deposits (Pine Creek Orogen, Murphy Inlier)

Placer deposits

Over half of the gold occurrences are gold quartz vein deposits.

Native gold is the main mineral and is commonly present as micron sized grains; coarse nuggets are rare.

Gold is commonly associated with pyrite, arsenopyrite and pyrrhotite and in places with minor base metal sulphides. Quartz, chlorite, sericite and carbonates are the common gangue minerals in the gold quartz deposits.

All gold deposits in the Northern Territory show some structural control at the regional and deposit scales, with most deposits within the Pine Creek Orogen trending northwest southeast. Base metal veins in the Pine Creek Orogen strike significantly differently than the gold veins, suggesting different discrete mineralizing events. They are interpreted to be syngenetic.

Most deposits show a preference for competency contrast situations in dilatant or low pressure zones, such as anticlinal crests, recurrent shear zones and necking zones. Gold mineralization is invariably late, occurring after orogenic events.

Common factors for most gold deposits include:

Gold deposits are nearly all in low grade, sub greenschist to lower greenschist facies regionally metamorphosed sediments (commonly greywacke siltstone shale)

Anticlinal hinges and shear zones are generally the most favourable loci

Subsequent to regional metamorphism and deformation, the metasediments were intruded by I Type granite and the gold mineralization are within the contact metamorphic aureole

Fluid inclusion data suggest the involvement of moderate to high salinity fluids in temperature range from 200 300°C

Stable isotope data suggest a magmatic/metamorphic origin of these fluids

Five main types of mineralization have previously been recognized within the Pine Creek Orogen. These include:

Sheeted and stockwork quartz vein systems located along major anticlinal hinges in the Mount Bonnie and Burrell Creek Formations and to a lesser extent, the Gerowie Tuff. Mineralization is hosted by carbonaceous or sulphidic host rocks (Woolwonga) or along zones of competency contrast between greywacke and shale (Enterprise, Union Reefs, Goodall, Alligator, Faded Lily, Howley, Big Howley, Yam Creek and Fountain Head) or dolerite (Bridge Creek). Axial planar quartz veins have been identified in some deposits (Enterprise and Woolwonga). Stratabound quartz reefs occur in most of these deposits, and may develop into saddle reefs along fold hinge zones (Enterprise, Union Reefs and Fountain Head);

Sediment hosted stratiform gold mineralization and quartz sulphide vein hosted stratabound gold mineralization in cherty ironstone and carbonaceous mudstones of the Koolpin Formation (Tom' s Gully, Cosmo Howley, Golden Dyke and Rising Tide) or the Gerowie Tuff (Brocks Creek);

Stratiform, massive to banded, sulphide silicate carbonate mineralization in the Mount Bonnie Formation (Mt Bonnie and Iron Blow);

Sediment hosted stratiform and stratabound gold mineralization in cherty, dolomitic and sulphidic shales of the Mount Bonnie Formation, with sheeted quartz sulphide veins (Rustler' s Roost); and

Sheeted or stockwork quartz feldspar sulphide veins hosted by Zamu/Maud Creek Dolerite sills (Maud Creek, Howley, Howley South, Bridge Creek and Kazi). Most gold mineralization in the Pine Creek Orogen occurs within the South Alligator Group, especially above the Middle Koolpin Formation, and in the lower parts of the Burrell Creek Formation. At Maud Creek gold mineralization is hosted by the Tollis Formation that represents the uppermost unit of the El Sherana Group and unconformably overlies the Burrell Creek Formation. Most of the fold associated deposits were probably formed during intrusion of granitoids such as the synorogenic Cullen Batholith and the Burnside Granite.

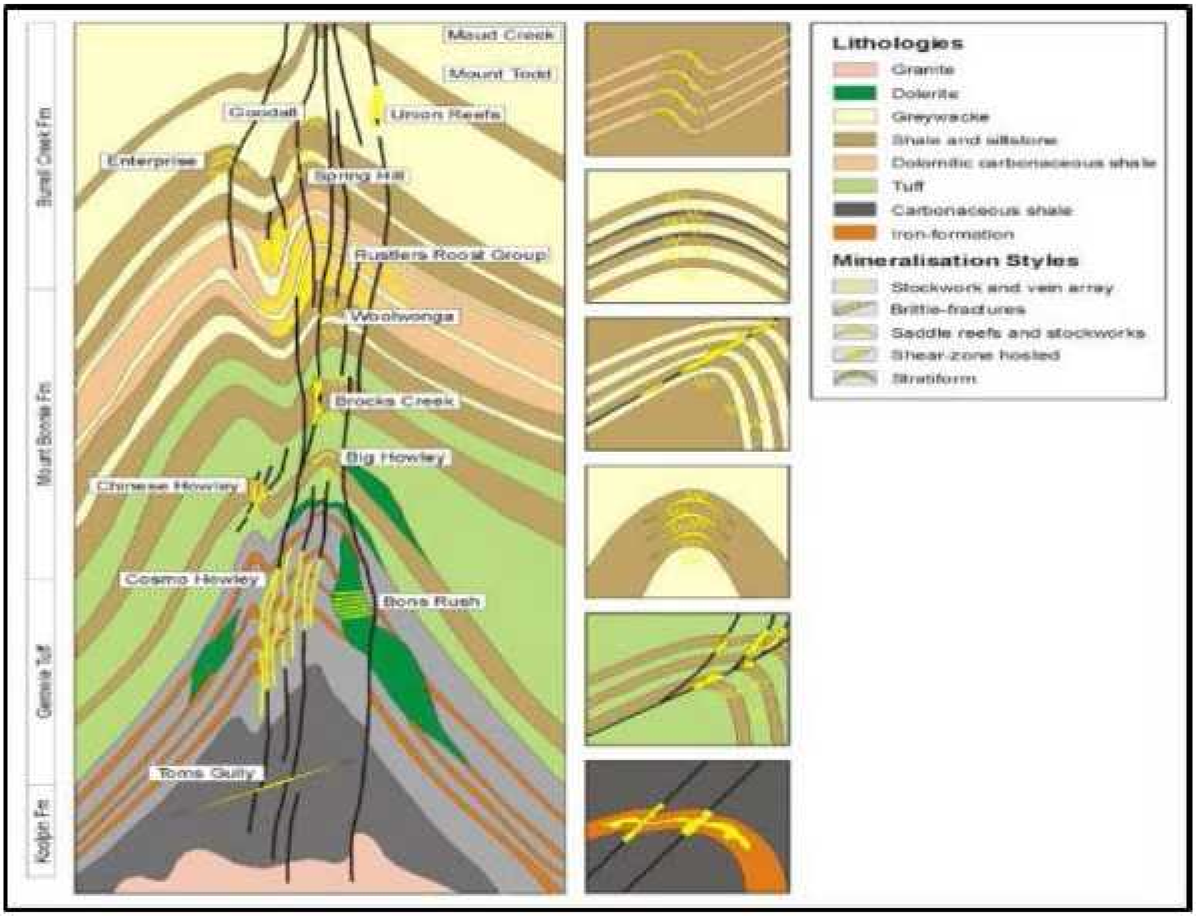
The most important regional scale exploration vectors to the orogenic style of gold mineralization are:

The position of the biotite isograd in the contact metamorphic aureole of the Cullen granitoids. The biotite isograd needs to be mapped out carefully in areas of exploration interest and exploration focused on the biotite albite epidote contact metamorphic zone

NNW NW oriented anticlinal axes appear to be the most productive. However, exploration cannot be totally restricted to anticlines in this orientation, as other anticlines or even synclines may be mineralized

Strongly interbedded and contrasting rock types (e.g., greywacke siltstone) particularly in the upper parts of the stratigraphy in the Mount Bonnie and Burrell Creek Formations in particular

Carbonaceous or iron rich lithologies in proximity to indications of gold mineralization. Such lithologies and any veins within them need to be mapped out carefully to help locate potential trap sites for economic gold mineralization.



Pine Creek Orogen (Sener, 2004)

Figure 8-1: Structural - stratigraphic model for Newmarket Gold deposits

8.2 Structural Models

Assuming that the majority of gold deposits within the Pine Creek Orogen are structurally controlled and mesothermal/orogenic (cf. Groves et al. 1998) in origin, it is likely that the known gold deposits are associated with regional shear zones and fault systems that were formed during orogenesis. By analyzing maps displaying total magnetic intensity (TMI) data, a number of continuous, NNW trending first order faults can be defined within the sedimentary dominated rock sequences of the Burnside tenement area (Figure 8 2).

The majority of known gold deposits within the tenement area are spatially associated with the first order, NNW trending shear zones. It is therefore likely that these first order shear zones acted as conduits for epigenetic gold bearing fluids during/after orogenesis and they control the distribution of gold mineralization known in the tenement area. Additional factors such as the presence of the South Alligator Group, proximal antiformal hinges (e.g., Cosmo Howley) or converging secondary shear zones (e.g. Crosscourse) would also play an important role in localizing gold mineralization.

The major shear zones are separated by rock sequences that regularly preserve NNW trending, doubly-plunging anti-formal hinges with no clear evidence for strike slip deformation along these NNW trending structures. South of the Burnside granite area, a series of NE trending shear zones and faults have also been defined (Figure 8-2). Based on preserved asymmetries of rock sequences either side of these NE trending faults, dextral dominated strike slip deformation possibly occurred along these relatively later structures.

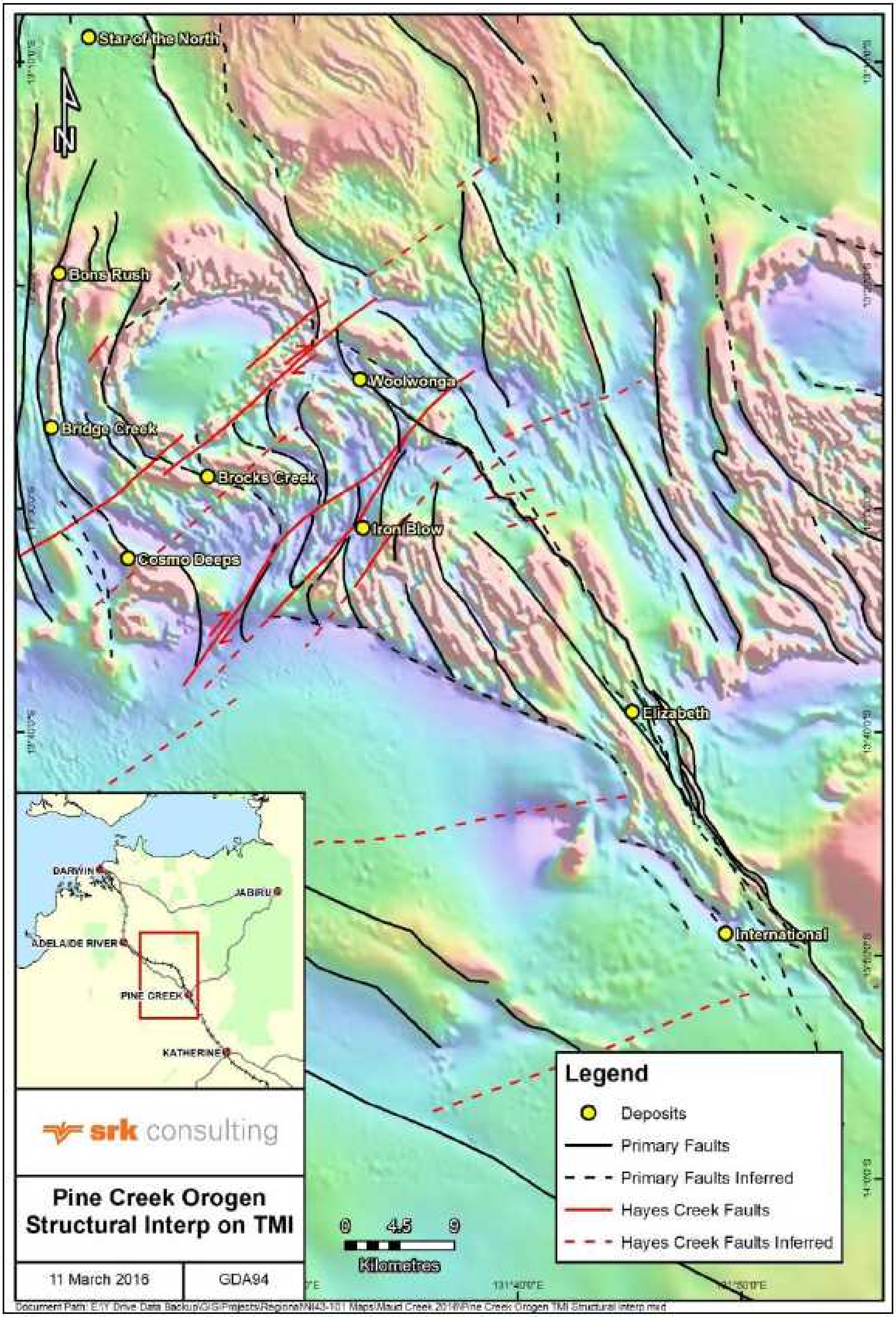


Figure 8-2: **Pine Creek Regional structural interpretation**

9 Exploration

9.1 VTEM Airborne Survey

A total of 590 line kilometers of VTEM survey were flown in the Maud Creek area (Figure 9-1) in 2011 covering an area of approximately 300 km^2. Line spacing was usually 200 m in the northern area (NW SE direction) and at 400 metre line spacing (NE SW direction) in the southern part of the area (Table 9-1). Southern Geoscience completed an initial interpretation and report of the survey data and Newmarket Gold geologists incorporating geology and available geochemistry reinterpreted this report (Figure 9-2).

Sixteen strong conductive targets and two moderately conductive targets were identified in the Maud Creek block. It should be noted that the strength and quality of conductors is significantly reduced from those defined at Burnside and Moline. The flat lying limestones and basalts that mask the underlying Proterozoic rocks likely play a significant role in the anomaly definition.

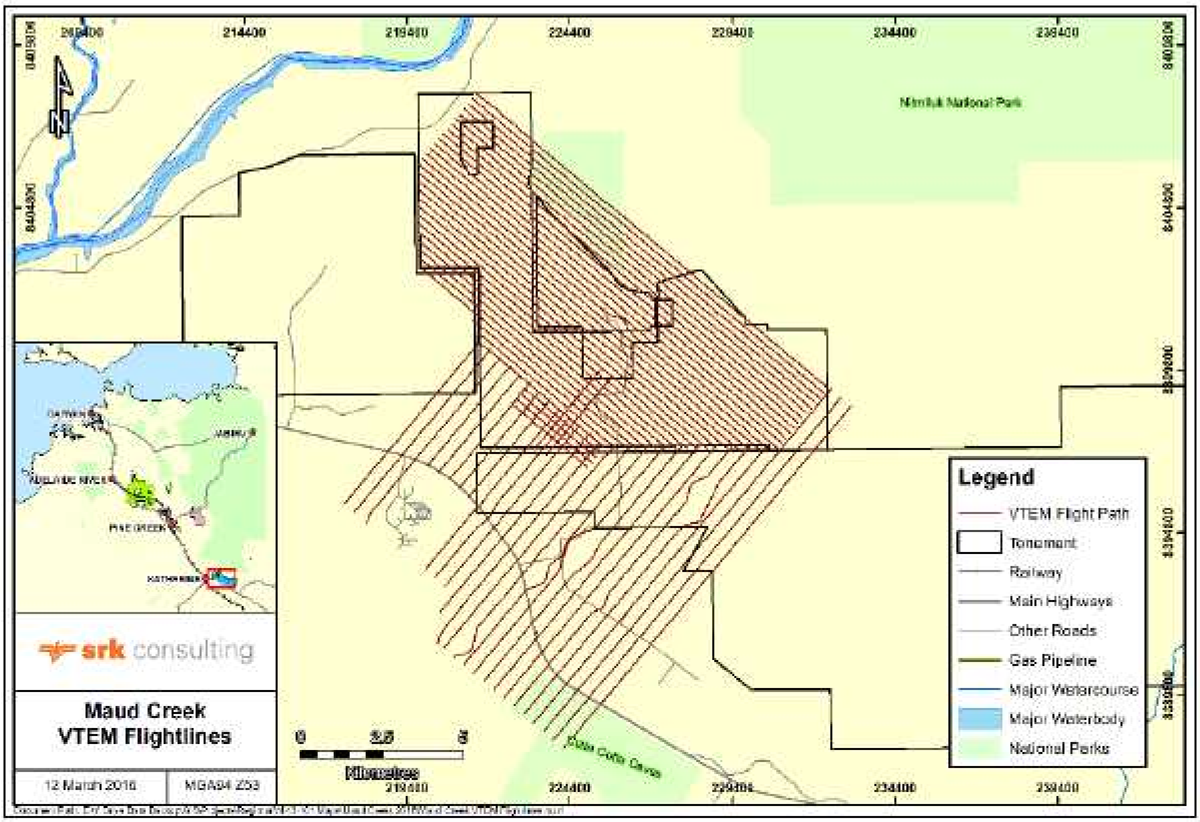


Figure 9-1: Maud Creek Property with VTEM flight lines

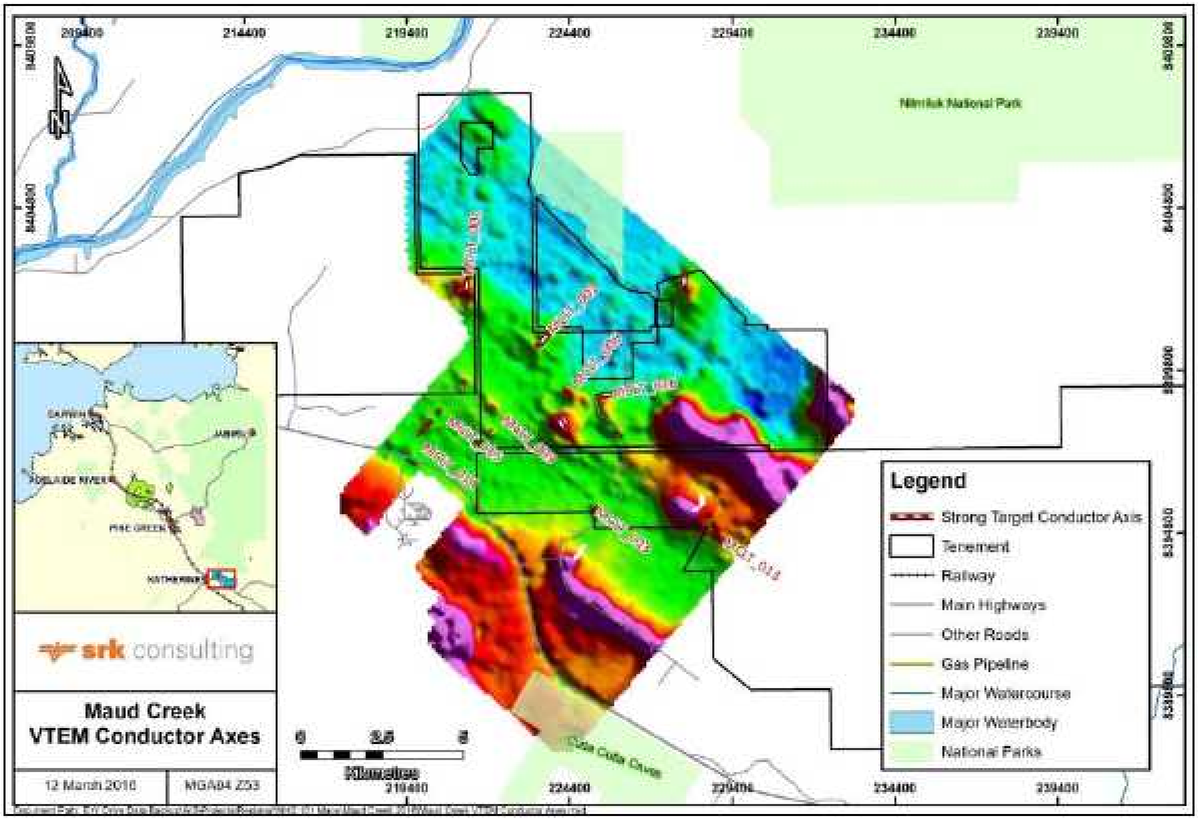


Figure 9-2: Strong VTEM Conductors on a 35 hertz conductor base map

The 25 m line spacing airborne magnetometer/ radiometric survey flown by Kalmet in 1997 clearly defines the extent of the younger Atrium basalts (Figure 9-3). Their signature produces a distinct noisy mottled effect on various manipulations of the magnetic data. The Maud Dolerites appear to have distinct magnetic anomalies along the margins of the body but the centre of the body appears to be magnetically quiet. It is quite possibly a differentiated intrusive.

Late, generally northeast trending dolerite dykes are readily apparent on the aeromagnetic image. Disruptions can be seen in these dykes indicating that later faults have created minor offsets.

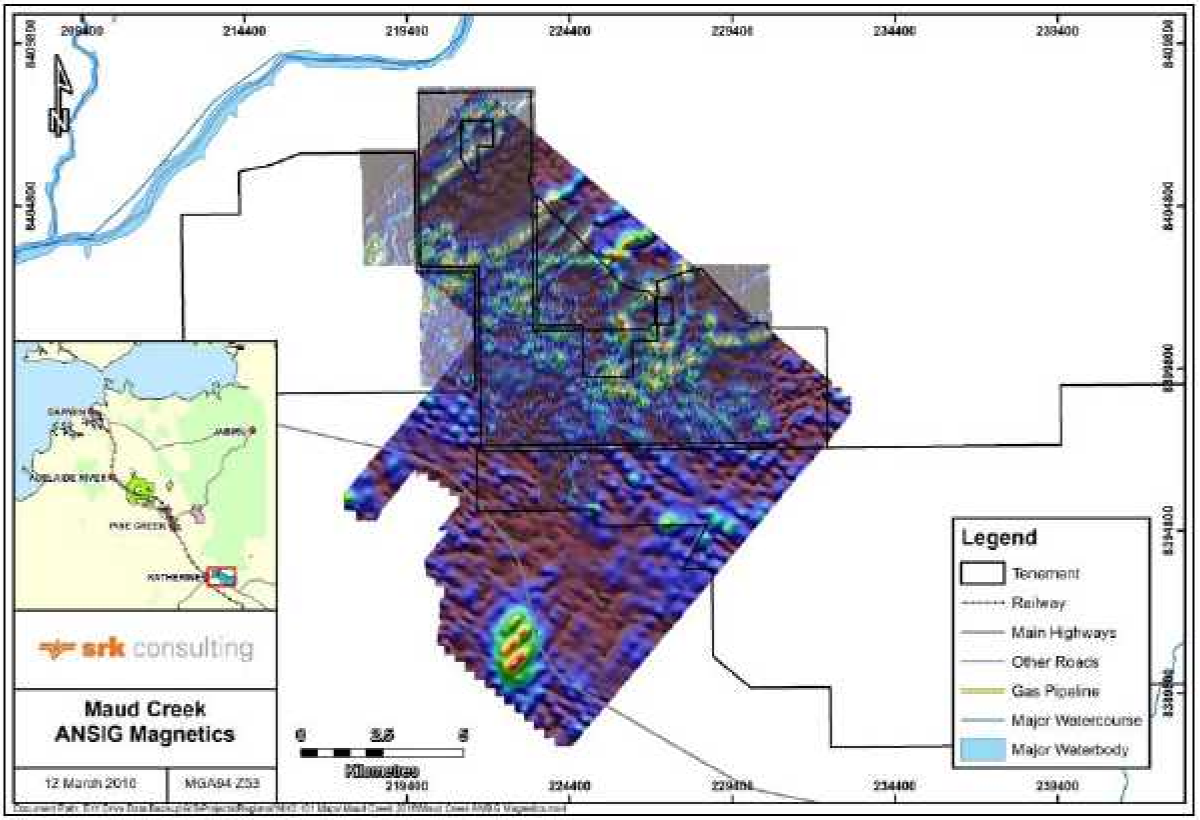


Figure 9-3: Merged 1997 and 2012 Aeromagnetic data with strong VTEM conductors (Card, 2012)

Table 9-1: VTEM conductor prioritization – Maud Creek survey area

Conductor #	SG Priority	Priority	Length (m)	Surface Work	Host Fm	Comments
MCLT005	1	1	400-600	No	lmstn	SW dipping, near a lmstn o/c, quartz veining noted in the area, previously Maxwell modeled, don' t have the model. Low ground between o/c. Deep seated. NW strike, no distinct magnetic signature. Weak Pb and Au in soil anomalies. Geochemistry likely masked by lmstn. May be on a NNW trending lineament
MCLT003	2	2	200	No	lmstn	Embayment in lmstn? Flanking linear magnetic anomaly to the north. Dyke? Right on the edge of High res mag survey area. Soil color anomaly on Google? Possibly on a north south lineament. Isolated conductor
MCLT007	3	2	200	No	lmstn	Just north of a stream, soil color anomaly, weak magnetic anomaly indicated on high resolution magnetic. Distinct direct magnetic anomaly. Work this first. Deep seated. Need to model
MCLT008	3	3	200	No	lmstn	Small magnetic anomaly, near o/c
MCLT009	3	3	200	No	lmstn	Small magnetic anomaly, larger anomaly to the NW, covered by high resolution magnetic. Near o/c to the north. Deep seated >200 m?
MCLT010	3	3	200	No	lmstn	Small magnetic response, on the exploration tenement. o/c to the north soil color anomaly to the north
MCLT011	3	3	200	No	lmstn	Small magnetic response, on the exploration tenement. o/c area
MCLT012	3	3	200	No	lmstn	Small coincident magnetic anomaly, on exploration tenement. o/c in area
MCLT014	3	3	200	No	lmstn	Small coincident magnetic anomaly, on exploration tenement. North of o/c
MCLT006	3	3	200	No	lmstn	Close to a NE trending lineament located to the SE. SE dip. Pb in soil anomaly
MCLT016	3	3	600-800	No	lmstn	SG says probably culture. On road but migrates to the south. South dip. Fence line but no different from other fences. Walked the road. Lots of road metal contamination from mine area. Limestone cover. o/c to south. Magnetic anomaly coincident on eastern line. Noisy data.

Conductor #	SG Priority	Priority	Length (m)	Surface Work	Host Fm	Comments
MCLT015	4	3	200	No	lmstn	Very similar to 007, 008, 009, 010, 011, 012, 014. Determine the cause of 007 first. Very small target. Non magnetic, o/c area. Edge of soil sample survey
MCLT002	4	3	800- 1000	No	lmstn	Possibly thick conductive overburden. On a north south lineament. No distinct magnetic signature. NE strike. On the NW trending base metal trend but striking the wrong direction
MCLT001	4	2	800- 1000	No	lmstn	Possibly conductive overburden. No distinct magnetic signature. On exploration tenement. Changes strike direction from ENE to NS. On base metal anomaly but strikes the wrong way. Low ground. Possible o/c area. Possible north south lineament with Chessmen and Red Queen to the north
MCLT004	4	1	200	No	lmstn	At the south end of a linear magnetic feature that hosts Maud Creek to the north. Also on a NW trending magnetic feature that may have an association with the base metal in soil anomaly. Watch out for buffalo. Increased vegetation in the area. Slight soil color anomaly
MCLT013	4	2	200	No	?	Very small and narrow, very isolated response. Weak magnetic anomaly. Possible east west lineament. Outside soil survey area.
MCLT017	4	1	600-800	Yes	Maud dolerite	At old Maud Creek workings. Change in strike direction to conductor. ENE to NNE. Associated with magnetic anomaly. Northern response outside property boundary. Highly conductive regolith. Alluvials in area
MCMT002	NP	1	200	No	lmstn	SE of a large magnetic anomaly interpreted to be in a regional NE trending structure. Appears to be in a nature park, karst caves. Interesting target but deep seated and too many social issues. Ignore

9.2 Stream sediment survey

In the mid 1990' s a fairly extensive stream sediment survey (approximately 173 samples) was carried out in the Maud Creek area. Samples were taken along various drainage systems at sample spacing of 500 m or closer. It is believed that a BLEG analysis was carried out with elements such as silver, copper, lead, zinc, bismuth, molybdenum and antimony also determined (Table 9-2).

In 2012 it was decided to stream sediment sample the 600 km2 area of EL 28902 in order to quickly determine its mineral potential. Arnhem Exploration Pty Ltd were contracted to carry out the survey.

A total of 164 stream sediment samples were collected along streams at approximately 1 km spacing. Figure 9 4 displays the sample distribution over the tenement area. Note that there is some duplication/overlap in the areas sampled with the 1996 97 survey. Samples were collected from several sites at any particular collection point in order to alleviate point anomaly sources.

Each site was photographed. Samples were sieved to 75 microns in the field. A total of 9 duplicate samples were taken for QAQC purposes.

A visual examination of the QAQC sample duplicates indicates that the gold values exhibit good repeatability while the other elements returned acceptable levels of repeatability. It should be noted that one sample exhibited marginal repeatability in lead and copper.

A correlation matrix of the 2012 survey data population indicates that there are weak correlations between gold and arsenic (0.38), bismuth (0.36), nickel (0.42) and tin (0.37) . There are significantly higher correlations with silver and copper (0.87), lead (0.88), antimony (0.92), arsenic (0.61) and barium (0.66) . It is suspected that several mineral deposit types are present in the area and that the correlation relationships are reflecting those differing styles.

A brief statistical look at the results for selected elements from the 1996 97 survey and the 2012 survey in Table 9-2 reveals the following:

Table 9-2: Comparative statistics between 1997 and 2012 soil survey results

Year	Mean Ag Grade (ppm)	Mean As Grade (ppm)	Mean Gold grade (ppb)	Mean Cu Grade (ppm)	Mean Pb Grade (ppm)
1997	0.021	4.1	0.699	33.1	55.2
2012	0.026	2.0	0.695	19.6	26.4

There is a good agreement between gold and silver values for both surveys. There is a wide divergence for copper, lead and arsenic. The base metal soil anomaly located south of the Maud Creek deposit skews the 1997 survey results. If both surveys were to be merged then the latter 3 elements would have to be normalized before they could be plotted and displayed effectively.

Underlying geology plays a distinct role when it comes to interpreting the stream sediment data. Largely Cenozoic and Mesozoic materials that overlay Proterozoic lithologies and/or Cambrian age sediments or volcanics underlie the significant land area to the east of the Maud Creek deposit. The masking effect of these younger sediments has likely diluted any anomalous effects from underlying Proterozoic rocks so subtle anomalies need to be field checked. It is conceivable that there are windows through the younger sediments that expose the Proterozoic as they do at the Copper Breccia occurrence that are located at the east end of the original Maud Creek tenements.

The gold results generated a number of anomalies. Most obvious is the cluster of anomalous samples in the Maud Creek deposit area and that area, which is underlain by Maud Dolerite. The 1997 survey did not produce an extensive gold anomaly in this area. It is suspected that sampling the finer sediments in 2012 produced more consistent results.

The 1997 survey produced a strong and extensive Au anomaly in the Red Queen/Chessmen area (northwest part of the property). This area was not re sampled in 2012 but the area to the southwest was and the gold anomaly appears to extend into this area.

In the area that the geology map indicates (Figure 9-4) there is extensive cover a number of gold anomalies occur. These all have anomalous multi element associations and all need to be ground checked.

Anomaly 1 is a cluster of two samples that are anomalous in gold, iron, chromium, arsenic, lead, molybdenum, tin and uranium. A preliminary interpretation of regional aeromagnetic data indicates that this anomaly may be situated within the southern limits of the Maud Dolerite unit (as defined by magnetics). SRK' s structural interpretation of the region carried out for Kilkenny Gold in 1998 indicates several large ENE trending structures that cut through this target area.

Anomaly 2 is one sample that is highly anomalous in gold (9.7 ppb) as well as molybdenum, bismuth and arsenic. The area should be resampled with either 3 or 4 stream sediments or a small soil grid established to cover the upstream area. The anomaly area does not exhibit any distinct magnetic signature

Anomaly 3 is a cluster of 4 samples that is anomalous in gold, copper, bismuth, arsenic, lead, chromium, uranium, tin, silver and barium. It is interpreted to be in the southern extension of Maud Dolerite. SRK' s structural interpretation of the region carried out for Kilkenny Gold in 1998 indicates several large ENE trending structures that cut through this target area. Anomaly 4 is located at the very northeast part of the survey area and is anomalous in gold, silver, iron, bismuth, arsenic, lead, chromium, uranium, tin and silver. The area is likely underlain by Tollis Fm. The regional magnetic data indicates it is proximal to a northeast trending dolerite dyke. Northwest trending structures can be interpreted. The gold in stream sediment anomaly appears to occur at the intersection of two major structures according to the SRK interpretation.

Anomalies 5, 6, 7, 8, 9 and 10 are one and two point anomalies. They all need to be ground checked. The extensive base metal anomaly defined from the Kalmet 1997 stream sediment and soil surveys is clearly defined on the normalized copper and lead stream sediment map. It would appear that is may extend over a distance of 30 35 km. It obviously weakens to the east and west but this may be a function of increased thicknesses of younger cover. The base metal anomalous area is also defined with elevated values in barium, cadmium, chromium, iron, potassium, magnesium, manganese, nickel, tin and zinc.

The base metal anomaly in stream sediments and soils doesn' t appear to be directly related to any individual VTEM anomaly and there does not appear to be a correlation to any particular magnetic response. The area of the soil anomaly is largely underlain by young Cambrian aged volcanics, although one area is underlain by Maud Dolerite. If the geochemical response comes from the Proterozoic rocks then there is some sort of mechanism allowing it to percolate through the younger volcanics. Looking at the airborne magnetic and radiometric data it would appear that the base metal anomaly occurs at or very close to the younger volcanic - limestone contact. The geochemical anomaly can be traced for >10 km in a generally east west direction.

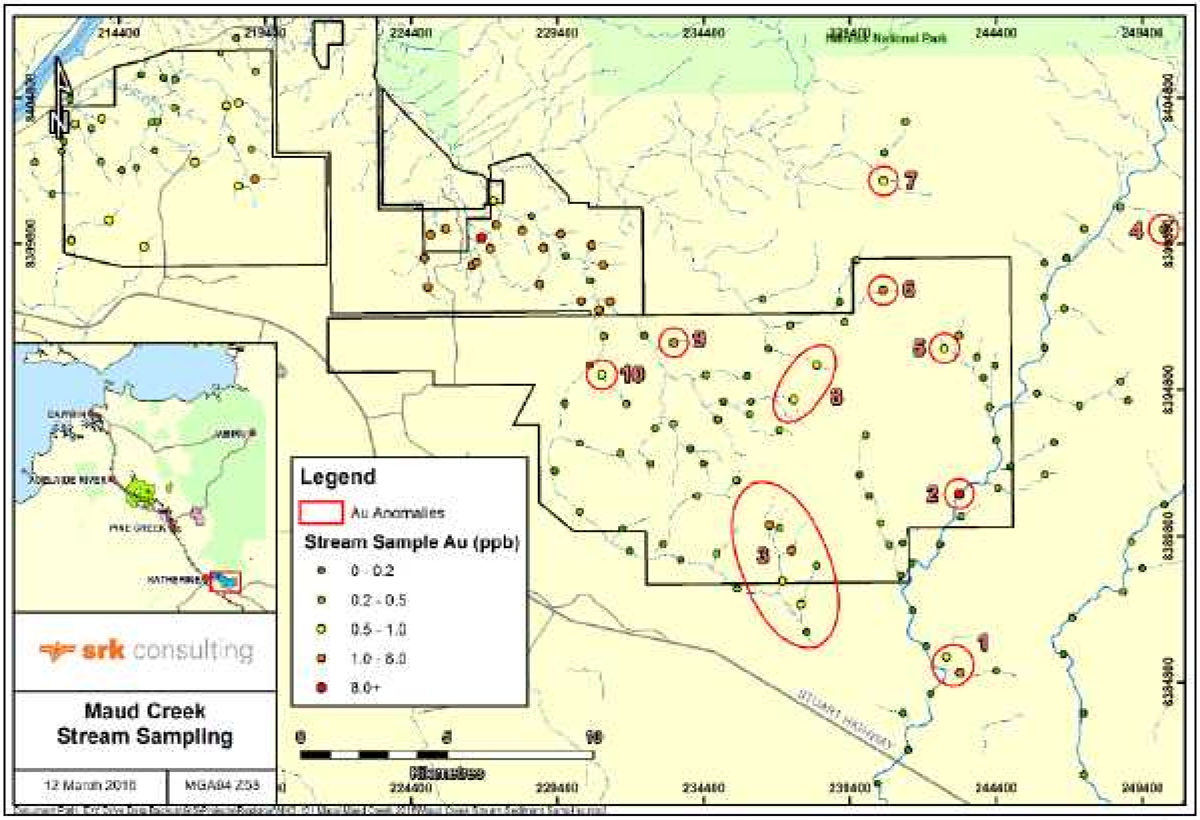


Figure 9-4: Maud Creek Area – 2012 stream sediment survey – selected gold anomalous areas

9.3 Soil sampling surveys

In 1997 Kalmet carried out an extensive soil survey over the Maud Creek area on lines 400 m apart and samples taken at 25 metre intervals but composited to 50 metre intervals (Figure 9-5). Samples were analyzed for gold, silver, copper, lead, zinc and antimony. A number of anomalous areas were defined but the coarse line and sample spacing didn’ t allow for accurate directional interpretations. Early in 2012 fourteen target areas were identified and given a letter designation. These did not include the VTEM anomaly reprioritization. The objective was to determine through rock chips, soil sampling and mapping the location of possible future exploration drillholes.

Prioritized Targets:

Anomaly A UTM 221500E 8404400N

- Two gold in soil anomalies. The eastern one is about 1200 m long with coincident antimony while the western one is 400 m long with coincident antimony over 800 m. Both have elevated arsenic values. It is interpreted that these are not overbank stream sediment related. No significant base metal anomalies.
- No significant VTEM anomaly associated with the geochemical anomalies
- The magnetic data indicates a possible NE trending dyke (late dolerite)
- Possible K anomaly. Distinct uranium/thorium anomaly,
- Possible north south structure interpreted from Total Count and SRK structural data.
- Chessmen and Red Queen occurrences to the north, possibly on the same north south structure.
- Likely underlain by Antrim volcanics, basalt

- Nothing distinct on Google image.
- May be the source area for the distinct and widespread gold in stream sediment anomaly in the area.

Anomaly B - UTM 221500E 8406400N

- Gold in soil anomaly over 1200 m running north-south with coincident antimony and arsenic over a 400 m length. There are streams to the north and south so there is a possibility that there may be contamination from overbank stream sediments.
- SRK structural geology indicates Tollis limestones and a north south structure, which may connect up with Target A.
- Very weak VTEM channel 25 response. Possible NNW trending feature.
- Magnetics indicate an interpreted NE trending dolerite dyke. Possible north south folds.
- Small but distinct uranium/thorium anomaly.
- Nothing distinct on Google image
- Need a compilation of past work at Red Queen and Chessmen prospects.

Anomaly C - UTM 221000E 8407400N

- The south part of the anomalous area has a stream close by so there may be overbank stream sediment contamination. One anomalous gold in soil sample to the west. There are elevated arsenic and antimony results to the north. Possible overbank contamination.
- No distinct VTEM response.
- Possibly the NW end of a NNW trending structure.
- Magnetics indicates a NE trending late dolerite dyke.
- There is a north south trending structure to the east.
- Distinct Total Count (TC) anomaly with some potassium contributing.
- Nothing distinct on Google image. Obvious stream

Anomaly D - UTM 222200E 8400000N

- Two anomalous Au in soil results. Soils here were not analyzed for arsenic, antimony or base metals.
- VTEM indicates a possible north-south structure on channel 25
- Likely underlain by Tollis limestone
- No distinct magnetic features
- Radiometrics indicates that the target is on the south edge of a major NW trending TC anomaly
- SRK map indicates possible volcanics at the edge of limestones.
- Nothing distinct on Google image.

Anomaly E - UTM 224000E 8401000N

- Gold in soil anomaly over 400-800 m running north south. Elevated arsenic and antimony. On the edge of base metal anomaly which is part of the major NW trending response.
- Kangaroo Flats prospect to the north. Don' t know anything about this prospect.

- Distinct magnetic anomaly, unknown cause.
- No distinct radiometric anomalies.
- SRK map indicates east- west structures
- Underlain by Antrim volcanics
- Nothing distinct on Google image-

Anomaly F - UTM 224300E 8402400N

- Gold anomaly in soils 400- 800 m north south. No significant arsenic, antimony or base metals. Possibly an overbank stream sediment anomaly.
- Kangaroo Flats prospect is 500 m to the west
- No VTEM response
- Radiometrics: possible NW structure and possible north- south structure. No distinct anomalies.
- Underlain by Antrim volcanics
- Low priority- target

Anomaly G - UTM 225000E 8401000N

- Two point gold in soil anomaly. No arsenic, antimony or base metals. May be overbank stream sediment.
- Near Curlies area to the SE
- Underlain by Antrim volcanics
- Distinct potassium anomaly
- SRK map indicates an east- west structure
- No distinct magnetic features

Anomaly H - UTM 225400E 8401600N

- Maud Creek deposit area anomaly
- Gold in soil anomaly extends 1600- 2000 m north- south
- Coincident arsenic and antimony with weak molybdenum and silver. No elevated base metals. Maybe contaminated from overbank stream sediments.
- VTEM weak channel 25 response trending north- south
- Sediment- tuff contact
- Antrim volcanics to the immediate west and south
- Radiometrics indicates a distinct north- south low as well as NE and NW lineaments.
- Magnetic low feature. West of Maud Dolerite.

Anomaly I - UTM 226000E 8402000N

- Gold in soil anomaly 600 m east- west x 500 m north south. Coincident arsenic and antimony anomalies. Quite possibly overbank sediments.
- Chlorite Hills to the NE.
- Surprise area between targets H and I.

- Underlain by Maud Dolerite
- North of NE trending late dolerite dyke
- Weak potassium anomaly - probably Maud Dolerite signature
- Close to Maud Creek and old workings-

Anomaly J - UTM 226700E 8401600N

- Gold in soil anomaly 400-800 m north-south trending. Coincident arsenic anomaly
- Antimony coincident over 1200-1600 m
- Jibaroo prospect.
- Maud Creek to the north. Possible overbank stream sediment contamination
- No VTEM response
- K/Th anomaly possibly caused by dolerite. NE structure indicated.
- North-south structure to the east.
- Good target area. Maud look-a-like target - geophysically
- East-west structure.

Anomaly K - UTM 227600E 8401600N

- Gold in soil low order anomaly with coincident anomalous antimony and strong arsenic. Fairly widespread so may be overbank contamination. Near a stream
- North - south structure to the east
- No distinct radiometric anomalies
- No distinct VTEM anomalies
- Magnetic low, magnetic destruction?
- Underlain by Maud Dolerite

Anomaly L - UTM 228200E 8401600N

- Runways prospect
- No streams in the immediate area.
- Gold in soil anomaly over 400-800 m with coincident arsenic and antimony.
- Base metals are very low order.
- Magnetic anomaly caused by Maud Dolerite
- Potassium/thorium anomaly interpreted to be caused by Maud Dolerite
- Bracketed by north-south structures.
- Good structural target. East margin of Maud Dolerite

Anomaly M - UTM 226400E 8399600N

- Small isolated gold in soil anomaly with coincident arsenic and weak antimony. Right on several streams so quite possibly overbank contamination.
- Drovers prospect, anomaly at south end of prospect

- Distinct north-south structure, good target area
- Intersecting NE and NW trending structures
- Potassium/thorium anomaly caused by Maud Dolerite.
- Linear magnetic low trending NNE

Anomaly N - UTM 227500E 8399600N

- Single point anomaly with coincident antimony and arsenic anomaly to the west.
- No proximal streams
- No distinct base metal anomalies
- Potassium/thorium interpreted north-south structure
- Maud Dolerite overlain by Antrim volcanics
- Possible NE trending magnetic anomaly interpreted to be caused by a dolerite dyke.
- No distinct VTEM anomaly

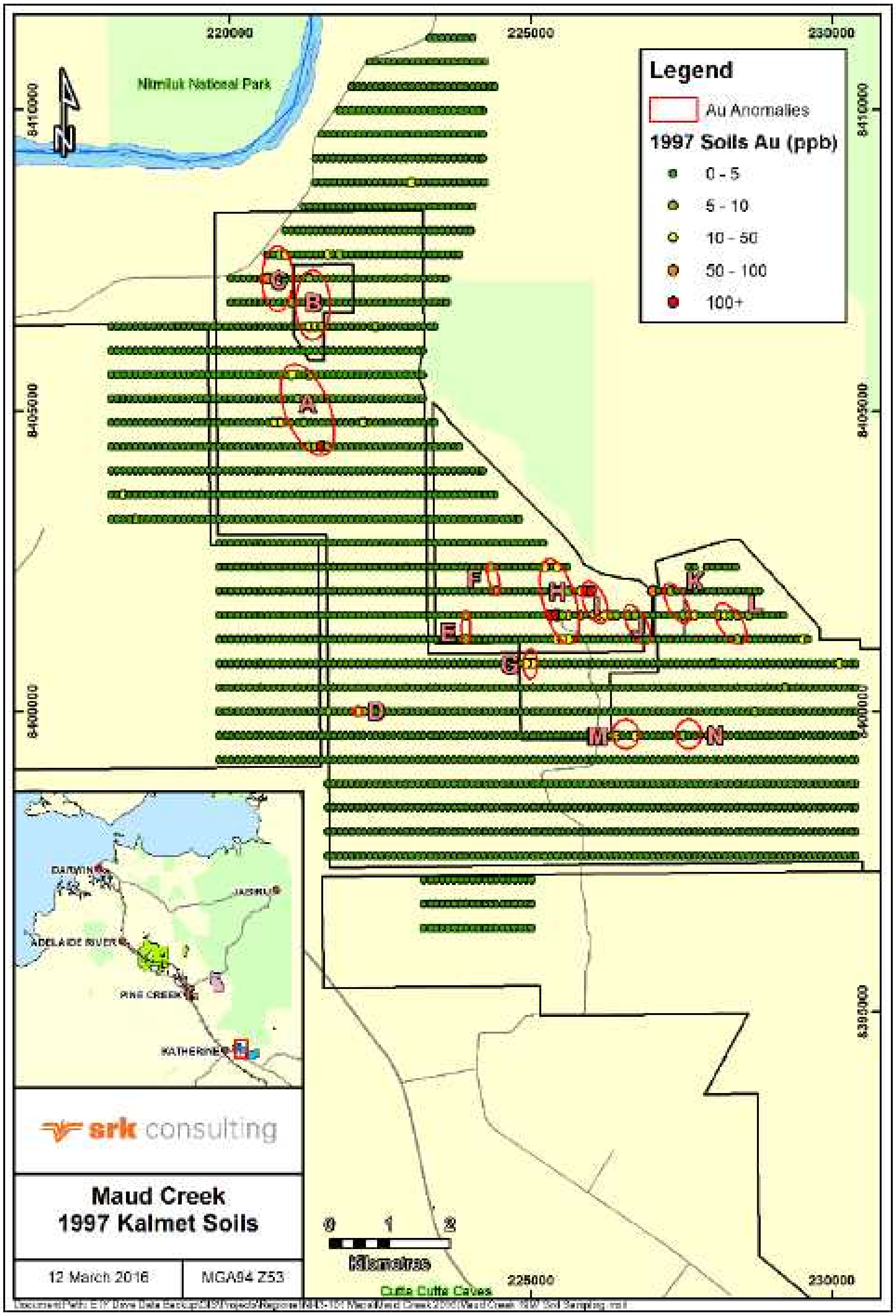


Figure 9-5: Location of soil anomalies requiring further work

A limited field program carried out later in 2012, designed to investigate the 14 target areas, defined the following:

Anomalies A and B

Anomalies A and B were originally contoured with a north‐south bias, however, field examination indicated that the anomalous gold‐in‐soil geochemical values should be‐ contoured in a north‐ northeast direction (Figure 9-6). When this was done two narrow gold‐in soil geochemical trends‐ with coincident arsenic and antimony anomalous values were defined, coincident with two 020 025° trending felsic dykes and/or silicification / brecciation.

Additional detailed soil sampling was recommended and ultimately carried out in 2012.

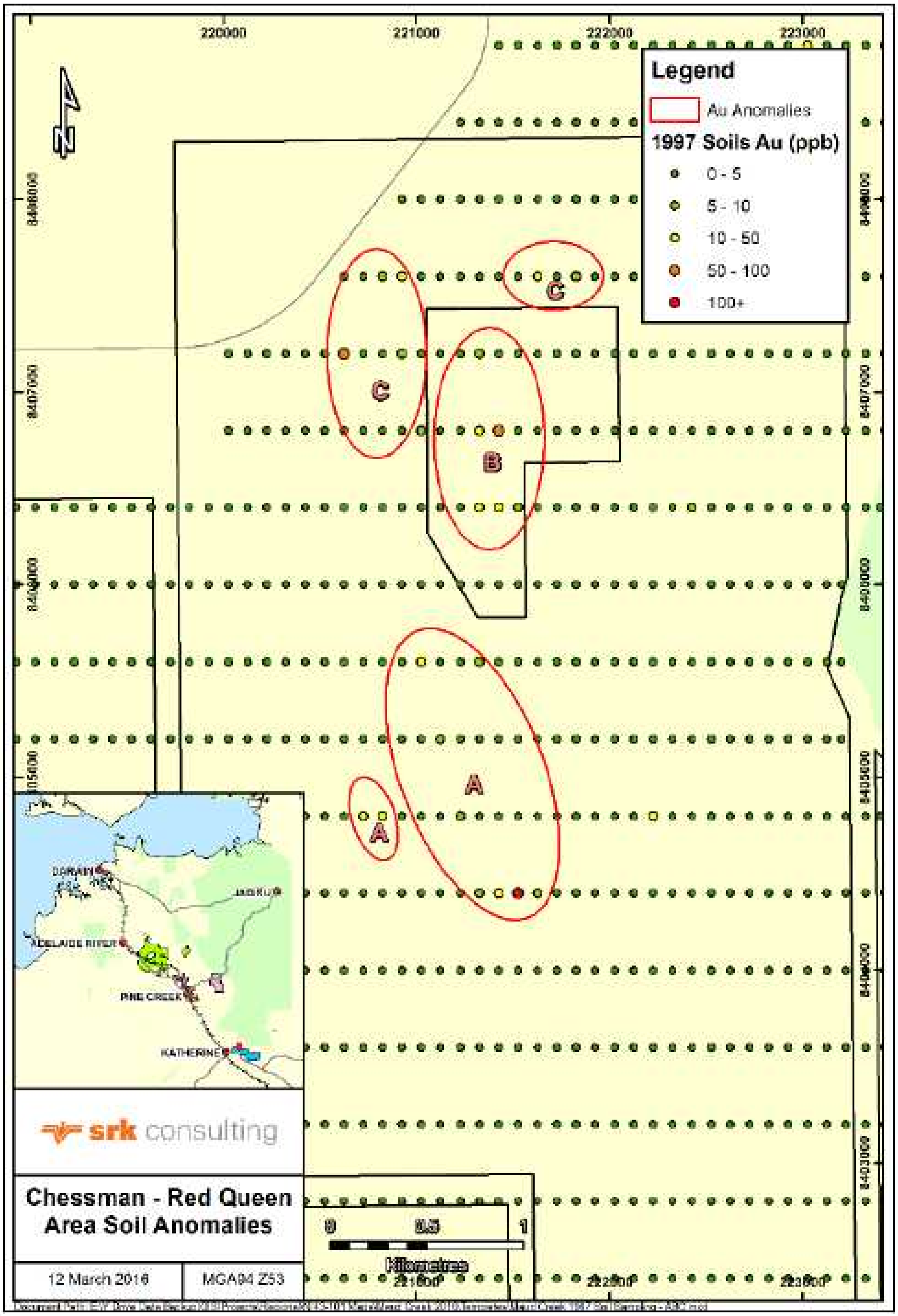


Figure 9-6: Red Queen - Chessman Area - Soil anomaly designation

The Chessman and Red Queen occurrences both occur on the northernmost trend (B) (Figure 9-6). The Chessman occurrence, located at UTM 8406600N; 221460E, is located south of a 58.6 ppb gold soil anomaly, and the Red Queen occurrence, located at UTM 8406230N; 221330E, south of a 11.3 ppb gold soil anomaly (both anomalies defined by past operators work). This siliceous zone on which both the Chessman and Red Queen occurrences are located continues southward for at least an additional 2 km, siliceous brecciated material located at anomalous gold-in-soil sample sites which returned 33.9 ppb and adjacent to soil sample site which returned 11.6 ppb.

Anomaly H

Anomaly H was only briefly visited as it defines the known mineralization at the Main Maud Creek- deposit (Figure 9-8). The strongest gold-in soil analytical results occur exactly on what is now the Maud Creek open pit mined in 2000. The northern and southern extensions of the gold-in-soil anomaly defined the mineralized trend of the Maud Creek deposit.

Chlorite Hills (Northern part of Anomaly I)-

The strong gold in-soil anomalous results at the north end of Anomaly I are probably defining the gold mineralization associated with the Chlorite Hills veining (Figure 9-7). Two fences of drillhole collars were located along a 25 x 25 metre grid spacing. Not all of these drillhole collars are in the Newmarket Gold database, however, those that are were drilled at a 270° azimuth, parallel to the observed quartz veining. Additional drilling completed by Kilkenny Gold was oriented in multiple directions, 320°, 140°, 180°, 90° and 270° azimuths. Anomalous gold, ranging from 0.54 g/t over 4 m to 2.25 g/t over 4 m, was intersected. The assembly of the drilling database for the Maud Creek Property area is still in progress.

There were no surface indications of a north south structure as indicated by Hill 50 NL in an internal memo by Bob Watchorn, however, the drilling completed by Kilkenny with 090° and 270° azimuths did intersect anomalous gold (up to 2.25 g/t over 4 m), possibly indicating the existence of north- south mineralized structures in the area.

A traverse was conducted between the Maud Creek open pit to the Chlorite Hills pits to try to determine if the mineralized quartz veins at Chlorite Hills extended to the Maud Creek deposit, however, there were no outcrop exposures until the Maud Dolerite. The contact between the Tollis Formation and the Maud Dolerite occurs at a north south oriented creek approximately 400 m east of the Maud Creek deposit. A second traverse was completed from the Chlorite Hills 040° trending pits ending exactly at the one drillhole completed at the Surprise area, possibly indicating- a 040° structure. (RC hole MCT-013 completed by Newmarket Gold 2011, 270° Azimuth, 60° dip. No significant gold analytical results were returned. The entire hole is in dolerite). There was no evidence noted to support previous interpreted north-south structures in this area.

O' Shea' s to Anomaly J

The O' Shea' s occurrence is located at UTM 8401920N, 0226920E. An outcrop of very fine grained siliceous rock disseminated with very fine sulphides with brecciated angular clasts to 10 cm occurs at the occurrence within coarse grained diorite. The silicified zone trends 30° with several pits excavated along its trend. Quartz vein mapping by Kilkenny Gold define NE trending veins in this area with additional minor NW trending veins.

The main workings strike north south with drilling completed east-west. A fair amount of historical drilling has been completed at the occurrence. RC drilling in 1998 targeted the NE trending shearing and associated workings. The mineralization intersected a 2 to 4 metre wide quartz/sulphide shear within the dolerite dipping 60° to 70° to the southeast.

A traverse at 30° was completed directly to Anomaly J (Jilleroo occurrence) gold-in-soil geochemical target arriving onto a series of shallow pits (Figure 9-7). These pits were investigating siliceous hematitic, highly altered rock and quartz veining and/or brecciation in hematitic diorite. Minor malachite was noted along fractures. Shearing and quartz veining were striking 135°. Two sub- parallel 135° trending shears were observed.

The Jilleroo occurrence may be occurring at the intersection of the continuation southward of the 30° O' Shea' s shear and the 135° structures.

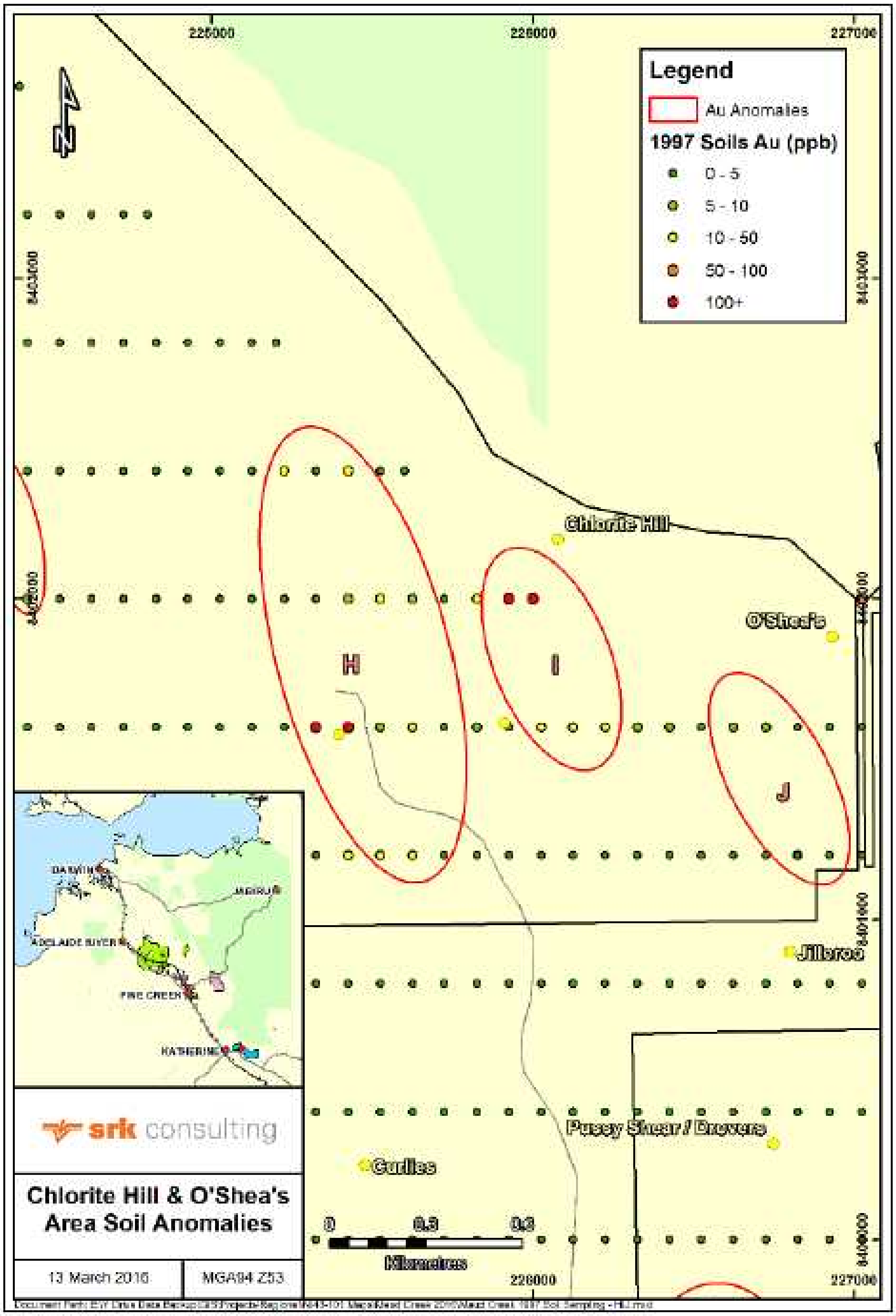


Figure 9-7: Chlorite Hills and O' Shea' s soil anomaly designations

Anomaly G, south end of H and south end of I to O' Shea' s single point gold- in- soil

Drawing a straight line lining up the eastern most anomalous gold- in- soil analytical result at Anomaly G (11 ppb) to the south end of Anomaly H (Maud Creek) (16 ppb) to Anomaly I (Surprise) (16.6 ppb) defines a 60° trending gold in soil anomaly (Figure 9-9). Extending this trend to the northeast the trend will pass immediately north of O' Shea' s and arrive at the Carpentania Pass target. The northeastern portion of this trend parallels a siliceous felsic dyke mapped in the area. This 60° trend has been noted numerous times throughout the property. The gold- in- soil analytical results may define a 60° crosscutting structure in the area or perhaps a fault that cuts the Maud Creek deposit at the southern end.

There has been extensive drilling completed (both RC and RAB) over the northern and southern extensions of the Maud Creek deposit by past operators. This drilling covered Anomaly G and Anomaly H. Very little drilling has been completed over the area east of The Maud Creek Deposit.

Pussy Shear Zone

A brief visit was conducted at the southern end of the Pussy shear zone area (Figure 9-8). Extremely hematitic gossanous material with a stockwork of cm wide quartz veinlets and sub-crops of agglomerate, similar to that which occurs at the Maud Creek deposit were noted. - A postulated north south shear zone was not confirmed. Additional prospecting should be completed along this trend to confirm Hill 50 NL's Internal Memorandum defining a north south structure. Several narrow 135° trending shears within diorite were noted. Several hand drawn sketches (completed by Kilkenny Gold) indicate en-echelon NW-SE trending structures within a north-south corridor. Grab samples collected by Kilkenny returned values between <0.01 to 0.18 g/t Au.

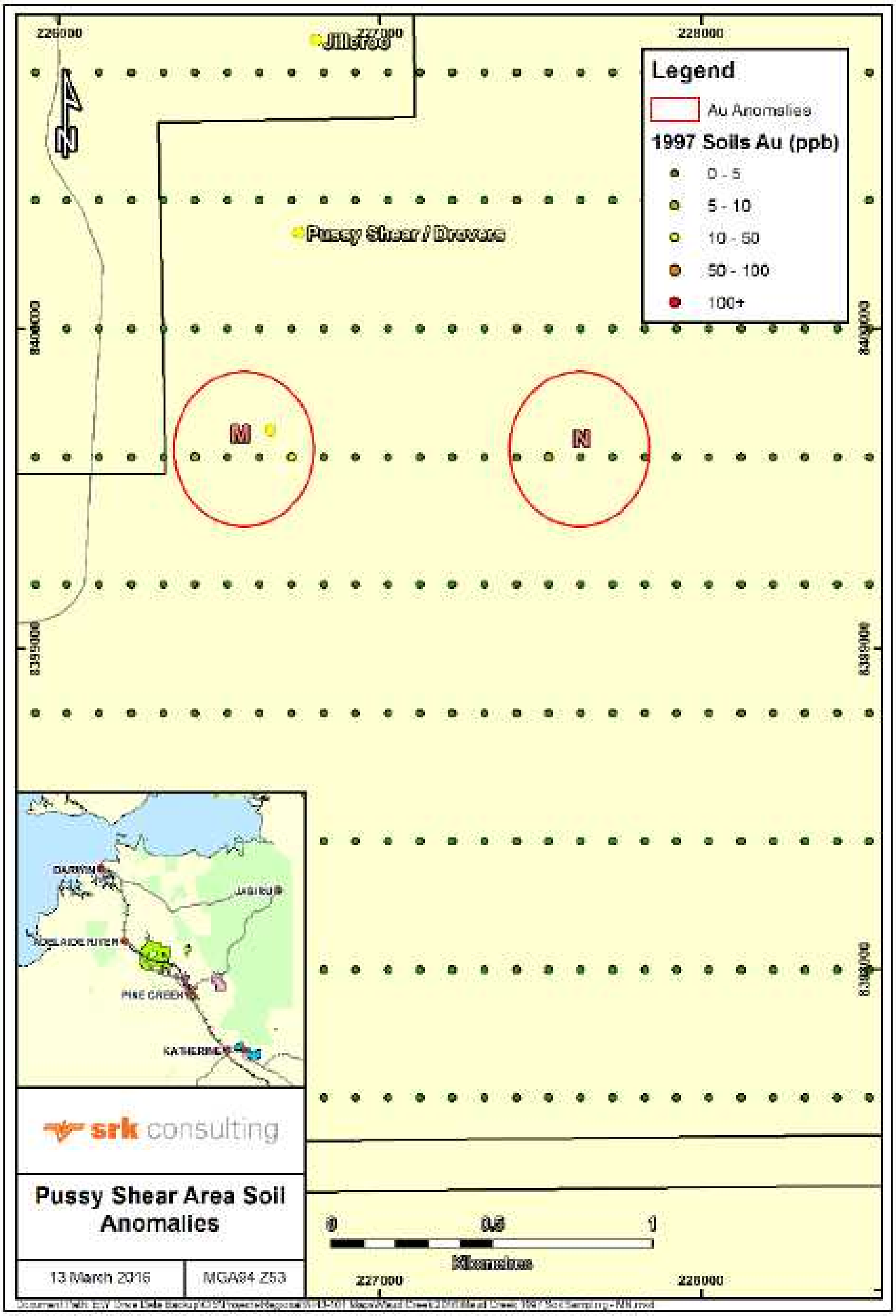


Figure 9-8: Pussy Shear zone soil anomalies

Kitten (Drivers) Anomaly

A brief visit was conducted at the Kittens (Drovers) Anomaly M area (Figure 9-9). A corridor of extensive quartz and quartz breccia- float trending 60° to 70° was located. Within this corridor several discontinuous siliceous sub crop ridges, massive to brecciated trending 80° were observed. In 1998 Kilkenny Gold drilled a fence of RC holes across the southern portion of this area (17 holes totalling 842 m). Drilling did not intersect any significant gold values. Two different intrusives were intersected by the drilling (best gold value was 0.8 g/t over one metre interval along one of the intrusive contacts).

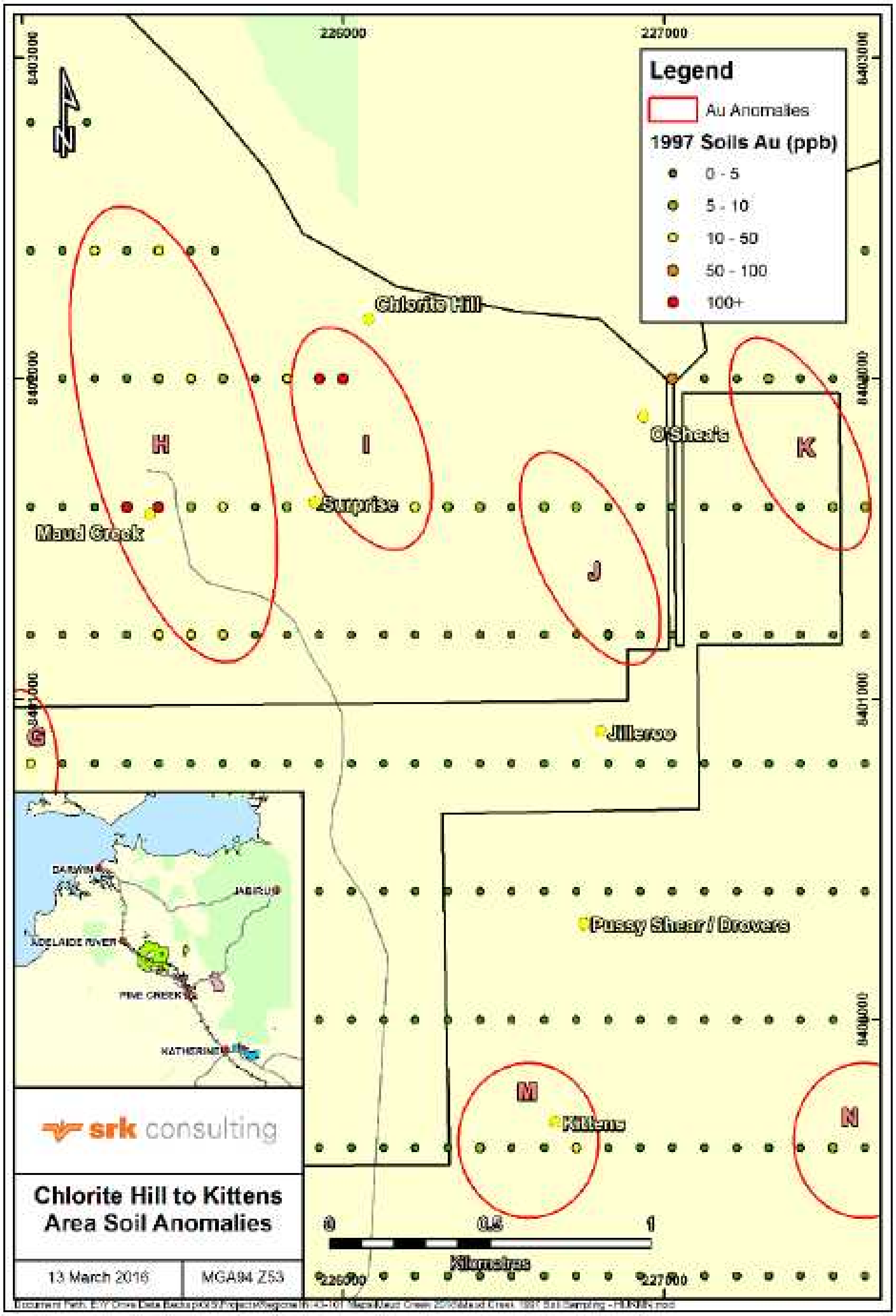


Figure 9-9: Chlorite Hills and Kittens (Droves) soil anomaly areas

Anomaly L

Gold in-soil Anomaly L occurs at the Runways target. The area is flat non-descriptive terrain with numerous float boulders of very fine grained dolerite within an area of reddish soil with patchy manganese coatings. - There were no obvious indications as to the cause or source of the elevated gold in-soil analytical results. Past operators completed two east-west lines of drillholes across this area, which extended across Anomaly K. A total of 44 RC holes (2,534 m) were drilled. The drilling returned no significant gold results (best value 0.14 g/t Au over 4 m). Buried intrusive and tuff were intersected. The NW trending magnetic anomaly in the area appears to correspond with these intrusives.

Anomaly E

Anomaly E (defined by two 7.6 ppb gold north-south correlated gold-in-soil analytical results) located on strike with VTEM anomaly MCLT_002 was prospected as part of the property examination. The area is flat, overlain with numerous boulders and rubble of quartzite, fine grained sandstone and chert. The overburden cover, as seen- in creek beds is at least one metre thick. There was no obvious cause for the elevated gold in-soil analytical results.

VTEM Anomaly MCLT_002

The area of VTEM conductor MCLT_002 is extremely flat. Minor outcrops of quartzite overlain by a thin cover of limestone were found. There was no obvious cause for the VTEM anomaly.

Subsequently, it was decided to soil sample two areas in detail, Chlorite Hills-O' Shea' s and the Red Queen/Chessman area. Samples were taken along lines 100 m apart, oriented to cross cut regional structures and geology. Samples were taken 25 m apart. The ionic leach method of analysis was selected so samples were taken at shallow depths of 10-20 cm and were sieved in the field to -75 microns. ALS Chemex' s Au + ME-MS41 0.0001 -0.1 Au by Aqua Regia with ICP-MS finish multi- element package was the analytical method used.

Chlorite Hills Area

In the Chlorite Hills/O' Shea' s area a total of 596 soil samples were collected by Arnhem Exploration with assistance from Newmarket Gold field assistants. This includes 20 sample duplicates taken to monitor QAQC of the commercial lab. These were subsequently shipped to ALS Chemex facility in Darwin NT. Instructions were to have the samples analyzed using Chemex' s ionic leach ME-MS23 multi-element package. A visual inspection of the 20 QAQC duplicate samples indicates that the repeatability for the sample population is good with no obvious errors.

A correlation matrix of the entire population indicates strong correlation (0.77) between gold and copper and a weak to moderate correlation between gold and silver and arsenic (0.35, 0.32) . Table 9-3 displays a comparison of the mean for a few elements from the two sample areas at Maud Creek and the one VTEM target area east of Bon' s Rush in the Burnside area.

Table 9-3: Comparative soil sample statistics for Maud Creek and Bons Rush Area

Area	Mean Ag Grade (ppb)	Mean As Grade (ppb)	Mean Gold grade (ppb)	Mean Cu Grade (ppb)
Chessmen	2.67	6.5	1.13	1,270
Chlorite Hills	8.9	4.97	5.35	2,249
VTEM Anomaly Burnside Area	3.41	13.43	0.25	975

The gold values in the Chlorite Hills area are significantly higher than those from the other two areas. The same applies for silver and copper (Figure 9-10). One explanation may be that the Chlorite

Hills/O' Shea' s area has seen historic mining with some significant ground disturbance and this contamination may be partially responsible for the elevated values in multiple elements.

Nevertheless, the gold in soil values exhibit an interpreted east west trend in gold that is not obviously repeated in other elements. The O' Shea' s and Chlorite Hills areas stand out as being quite anomalous in gold and somewhat in copper. O' Shea' s is anomalous in silver and antimony.

The west side of the grid displays anomalies in silver, calcium, iron, magnesium, nickel and anomalously low in zirconium, uranium and thorium. It is suspected that this area represents a contact with Maud Dolerite, which is interpreted from magnetic data to be a differentiated intrusive unit.

The eastern margin of the grid displays anomalism in calcium, magnesium and nickel. It is suspected that this exhibits a contact with a phase of the Maud Dolerite unit.

An anomalous area at the southeast end of the grid centred on 226900E 8401400N displays elevated values in a number of elements including antimony, thorium, titanium, uranium, zirconium, lithium, iron, arsenic, copper and weakly in Au. The area needs to be ground checked.

It is suspected that multi-element anomalism in the region of 226300E 8402200E may be due to overbank sediments associated with Maud Creek, which is situated immediately to the northeast. This needs to be ground checked.

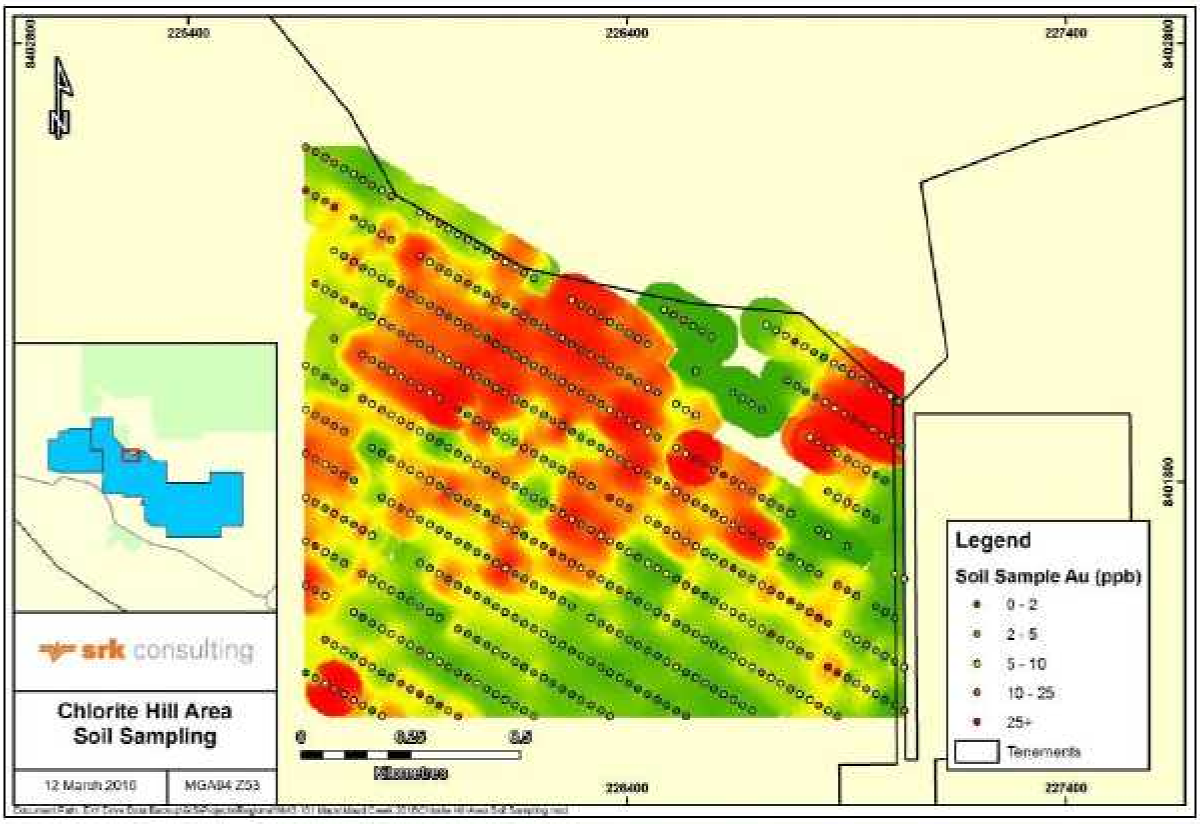


Figure 9-10: Au (ppb) for ionic leach soil results at Chlorite Hills, Maud Creek

Chessman Area

In the Chessman/Red Queen area Arnhem Exploration collected a total of 1,942 samples with assistance from Newmarket Gold field assistants (Figure 9-11). This includes 61 sample duplicates taken to monitor QAQC of the commercial lab.

A visual inspection of the 61 QAQC duplicate samples indicates that the repeatability for the sample population is good for most elements with acceptable variability between duplicate pairs. For gold values four of the duplicate pairs exhibit a variability that is outside acceptable limits. The other elements return acceptable values for the most part.

A correlation matrix of the entire population indicates a fairly strong match of 0.67 between gold and silver. The gold in soil values presents a unique distribution that is not matched with any other element. In Figure 9-11 a NNE trending anomaly extends for 2.5 km along the west side of the grid that would appear to extend beyond the grid to the NNW. A preliminary interpretation would indicate that the gold anomaly sub-parallels the west contact of a lithological unit that appears to be associated with a NNE trending syncline (graben?). In all likelihood its emplacement is structurally controlled. There is no distinct magnetic correlation with the gold anomaly although it would appear to cut through a strongly magnetic NE trending dolerite dyke located at the NW end of the grid.

Another Au in soil anomaly at the north end of the grid needs to be investigated on the ground. A highly folded magnetic anomaly located at the south end of the grid can be traced for many kilometers to the southeast. Malachite staining was noted in the area. The magnetic anomaly is coincident with contorted anomalies in copper, silver, magnesium, barium, calcium and lead and possibly a weak gold response. It is markedly negative in molybdenum, barium, tin, antimony, uranium and iron. Conceivably, this anomalous situation correlates with other significant base metal anomalies defined at the southeast end of the Maud Creek property. Further field investigation is required.

A variety of other elements are obviously defining differing lithologies that define the synclinal structure. An example is iron that defines the east and west margins of the syncline. It, along with a number of other elements also defined some folded structures within the core of the syncline. Field investigation is required to link these anomalous situations to specific rock types. The centre of the syncline has its own distinct geochemical signature and the ionic leach data along with magnetic and radiometric information can be used with great effect to help with geological and structural interpretations.

Calcium and magnesium data can be used to interpret crosscutting (NW SE) trending dykes. The VTEM data has defined a NW trending feature that correlates directly with a fence. A stronger NW trending conductive feature at the north end of the soil grid is likely associated with a dyke- like feature.

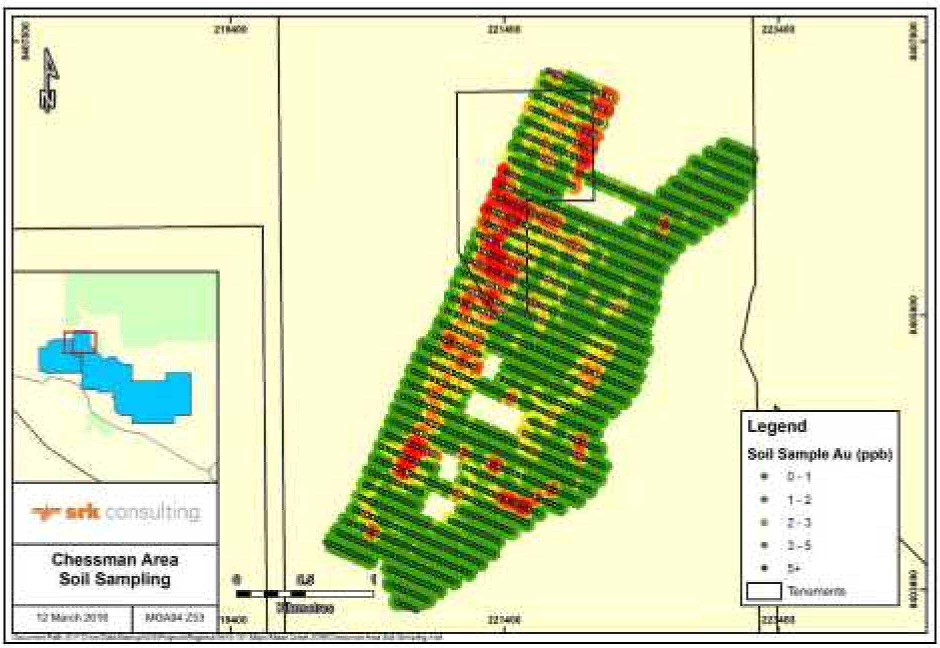


Figure 9-11: Ionic leach Au (ppb) in soil results, Chessman Area, Maud Creek

9.4 Rock Chip Sampling

During the reconnaissance prospecting of various soil anomalies a series of 58 rock samples were collected in the Red Queen/ Chessman and Chlorite Hills areas. Figure 9-12 displays the gold results.

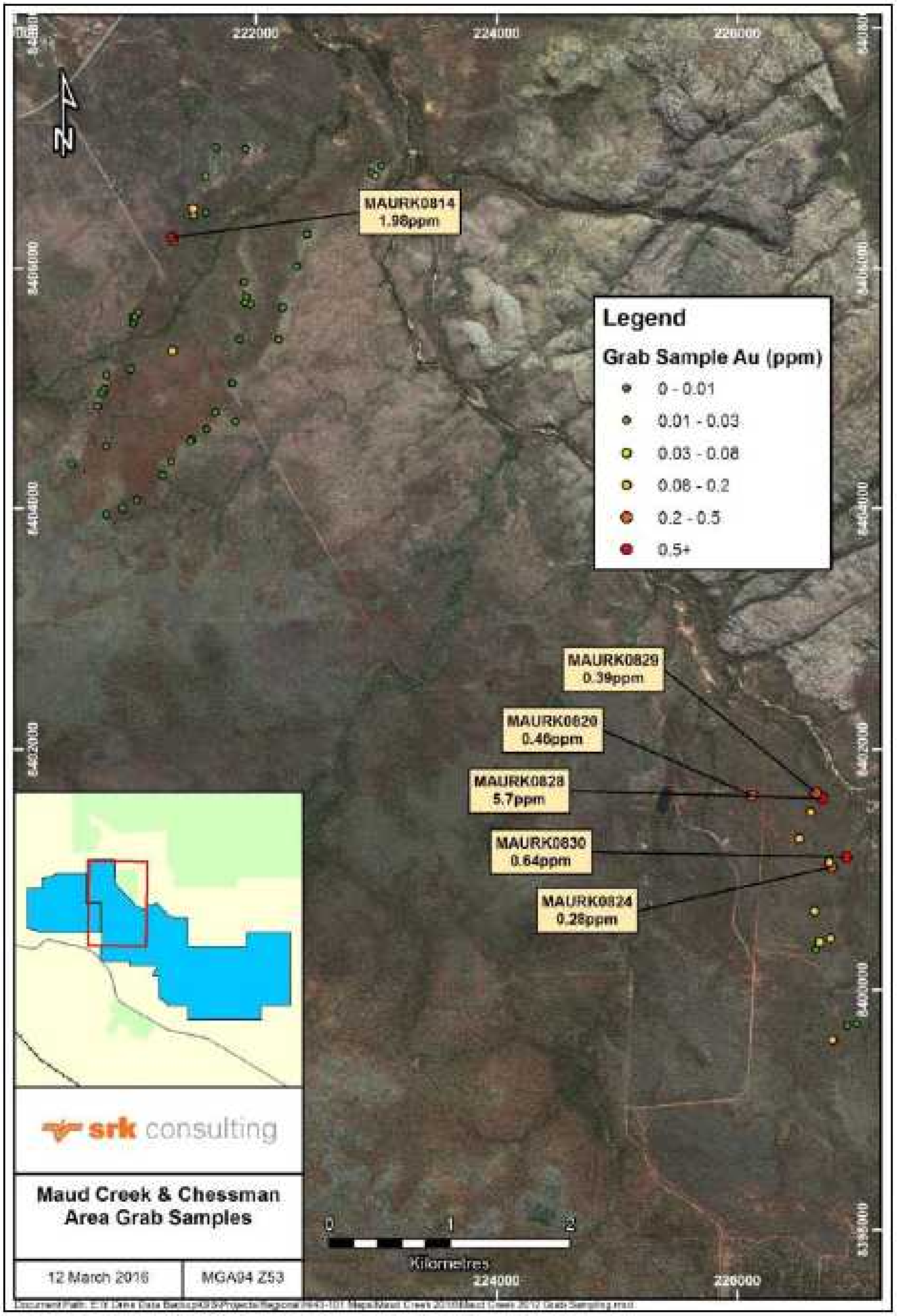


Figure 9-12: Maud Creek area gold in 2012 rock chip sample results

Anomalous gold values also display elevated values in silver, arsenic, bismuth, copper, iridium, lead, antimony, tin, tungsten and zinc. There is a distinct phosphorus depletion associated with the anomalous gold values. In the Red Queen/ Chessman area anomalous gold in rock chip values are Iocated at the north-west margin of the syncline where the soil results are anomalous. At the Chlorite Hills/O' Shea' s area anomalous gold in rock chip samples seem to be associated with the eastern margin of the Maud Dolerite.

10 Drilling

Over 94,000 m of RAB, RC and Diamond drilling has been completed at the Maud Creek Project and surrounding areas (Table 10-1 and Table 10-2). While the drilling prior to 2011 was not completed by Newmarket Gold, a significant amount of historical data is available for review and reporting. Based on the quality of the data available, SRK see no indication that the drilling information cannot be used for Mineral Resource estimation.

Table 10-1: Drill statistics for the Maud Creek deposit

		Diamond		RC	
Company	**Period**	**# of holes**	**Meters**	**# of holes**	**Meters**
CGAO	2011	6	3,180.33	14	702
Terra Gold	2007	1	211.50	0	0
Kalmet/Hill 50/Terra Gold	1995-97, 2001-2002, 2005	70	19,800.12	0	0
Kalmet	1994	0	0	356	42,390.8
Kalmet	1993	0	0	36	2,212
Placer	1990-91	20	2,992.37	0	0
Placer	1989-91	0	0	36	3,503.8
Total		97	26,184.32	442	48,808.60

Table 10-2: Historical drilling by previous tenement holders

Prefix	# of holes	Type	Company	Period	Notes
DRC	44	RC	KGNL		
GRC	17	RC	KGNL		
KR	282	RAB	KALMET		
MC01 - MC25	25	DR	MCML	Dec 1993	
MCE001 - 021	21	RC	KGNL		
MCP037 - 301	265	RC	KALMET	Sep 1994 - Jun 1996	
MCP302 - 416	115	RC			
MCP417 - 488	40	RC	HILL 50	Aug 2001 - Aug 2005	
MCW	54	RC	KGNL	Apr 1998 - Nov 1998	
MD001 - 033	33	DD	KALMET	1995	
MD034 - 052	21	DD	KGNL	1997	Includes 2 wedge holes by HILL 50 (2001).
MD053 - 062	14	DD	HILL 50	Aug 2001 - Aug 2002	
MD063 - MD065	2	DD	TGML	Jul 2005 - Aug 2005	
MRB	216	RAB	KALMET	Aug 1997 - Sep 1997	
MRC	36	RC	KALMET	Jan 1993	
RAB	588	RAB	MCML	Oct 1993	

SRRC	14	RC	KGNL	May 1998	
SWM	9	RC	KALMET	Jun 1998 - Sep 1998	
TMCD	1	DD	TGML		
WD	20	DD	PLACER	Jun 1990 - Jun 1991	Includes diamond tails to WP holes
WP	40	RC	PLACER	Jun 1989 - Jun 1990	

10.1 2011 Drilling program

Newmarket Gold, then trading as Crocodile Gold conducted infill drilling at Maud Creek in 2011: **Diamond** - Some holes were collared using PQ (122.6 mm) to allow for better recovery and to prevent collar collapse, this was then reduced to HQ once ground conditions allowed (Table 10-3). Diamond drilling was used at Maud Creek by Newmarket Gold (Figure 10-1) to ensure accurate logging of structures, lithology, alteration and mineralization, as well as capture of geotechnical data. Several diamond drillholes were also used for metallurgical sampling.

Table 10-3: Parameters used for infill diamond drilling at Maud Creek

Deposit	Diamond			RC			First drilled	Last drilled
	Holes	Meters	Size	Holes	Meters	Size		
Maud Creek	6	3,179	HQ-PQ	14	700	5”	19 Sept 2011	16 Nov 2011

Reverse Circulation - RC drilling was used in areas where diamond drilling was not required or appropriate, for example, to test the potential mineralization to the south of the Maud Creek pit where limited drilling was identified.

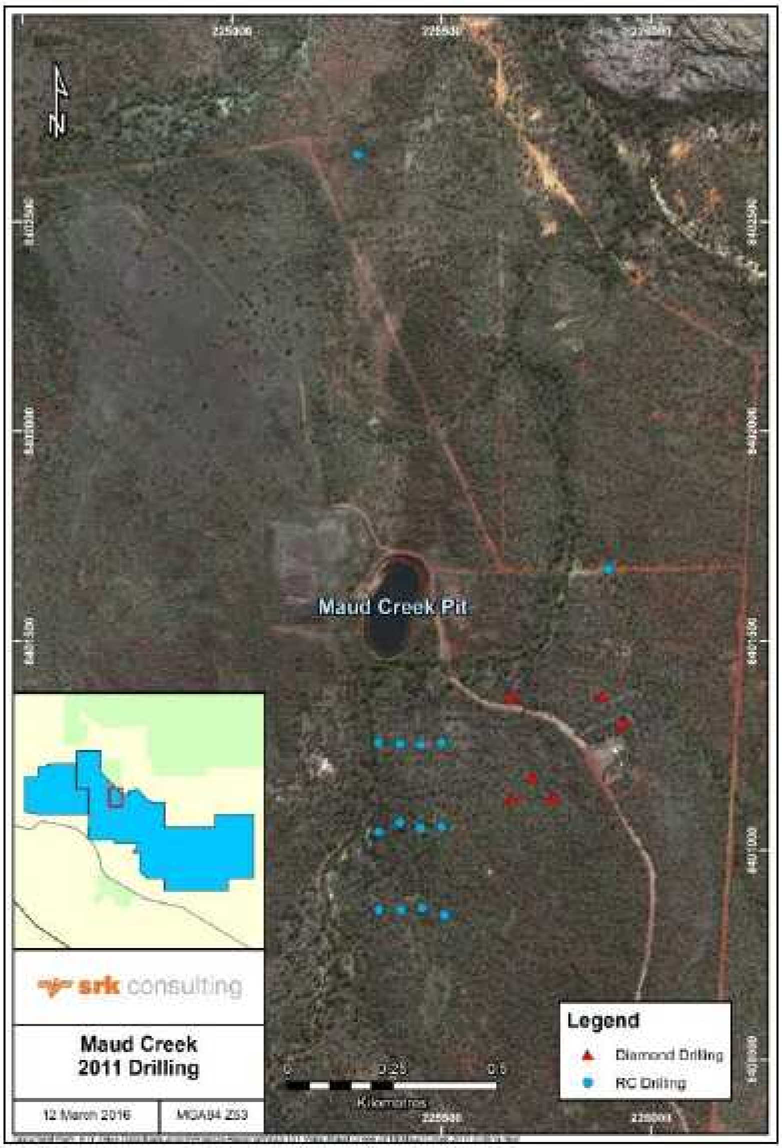


Figure 10-1: Plan view of the RC and DDH Drilling completed in 2011

The objective of the drilling program was to test the down plunge component of the Maud Creek deposit at depth. Four holes were used to test for deposit extension to the south, while two were used for verification of the resource model (Figure 10-2 and Table 10-4).

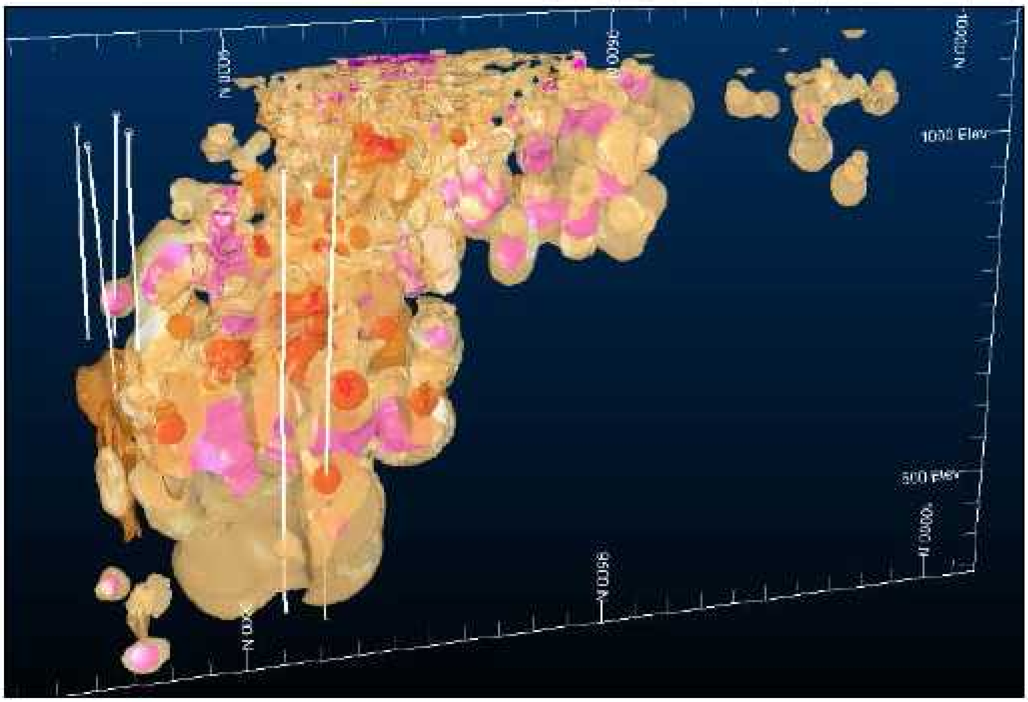


Figure 10-2: Long section looking west showing grade shells and 2011 diamond drillholes

Table 10-4: Maud Creek 2011 drilling data

Hole ID	X Collar	Y Collar	Z Collar	Depth	Type	Azimuth	Dip
MC001	19550	8799.996	1130	339	DD	275	-60.0
MC002	19500	8849.996	1130	372.6	DD	271	-59.7
MC003	19650	8799.999	1130	414	DD	279	-60.0
MC004	19600	8849.999	1130	483.6	DD	275	-60.0
MC005	19820	8974.997	1130	756.6	DD	270.9	-60.8
MC006	19770	9039.992	1130	813.5	DD	270	-60.1
MCRC001	19228.856	8539.006	1130.685	50	RC	275	-60.0
MCRC002	19281.995	8537.7	1130.949	50	RC	275	-60.0
MCRC003	19332.654	8542.016	1131.567	50	RC	275	-60.0
MCRC004	19384.094	8523.776	1132.461	50	RC	275	-60.0
MCRC005	19232.577	8725.36	1129.345	50	RC	275	-60.0
MCRC006	19282.385	8746.099	1130.508	50	RC	275	-60.0
MCRC007	19330.733	8732.725	1131.416	50	RC	275	-60.0
MCRC008	19381.4	8733.986	1131.779	50	RC	275	-60.0
MCRC009	19235.469	8937.985	1128.153	50	RC	275	-60.0
MCRC010	19286.715	8933.594	1128.934	50	RC	275	-60.0
MCRC011	19335.354	8930.858	1128.718	50	RC	275	-60.0
MCRC012	19384.542	8936.011	1128.761	50	RC	275	-60.0
MCRC013	19789.85	9342.812	1124.795	52	RC	275	-60.0

MCRC014	19210.585	10341.349	1124.459	50	RC	275	-60.0

10.1.1 Surveying

All holes were either surveyed by single shot and/or gyro survey. All drillhole collars were picked up by surveyors, and some historical drill collars were also resurveyed.

10.1.2 Core Recovery

Core recovery was not recorded for the Maud Creek 2011 drilling.

10.2 Sampling prior to 2011

Prior to the then Crocodile Gold' s acquisition, four companies had previously run exploration programs at Maud Creek, utilising both Diamond (23,004 m) and RC (48,107 m) drilling techniques (Table 10-2). A summary of the associated procedures was carried out by Snowden in their 2006 report Addendum to the Technical Report entitled Independent Technical Review of the Burnside, Union Reefs, Pine Creek and Maud Creek Gold Projects, Northern Territory, Australia - Resource Update, Maud Creek Gold Project. This report summarised any available historical data from Maud Creek produced by Placer Dome, Kalmet, Hill 50 and Terra Gold. GBS Gold requested the review by Snowden in 2006, but never conducted any drilling programs at Maud Creek during their tenure.

10.2.1 Surveying

As part of the review, GBS contracted a professional surveyor to locate all drillhole collars.

GBS also utilised the following downhole survey instruments to verify the orientation of the drillholes:

- Sperry Sun system
- Single shot system

Measurements were typically taken every 25 m.

Figure 10-3 and Figure 10-4 highlight the changes in azimuth and dip with drillhole depth as captured by GBS. The drillhole azimuths typically wander +/- 15°, with azimuth showing a relationship with depth.

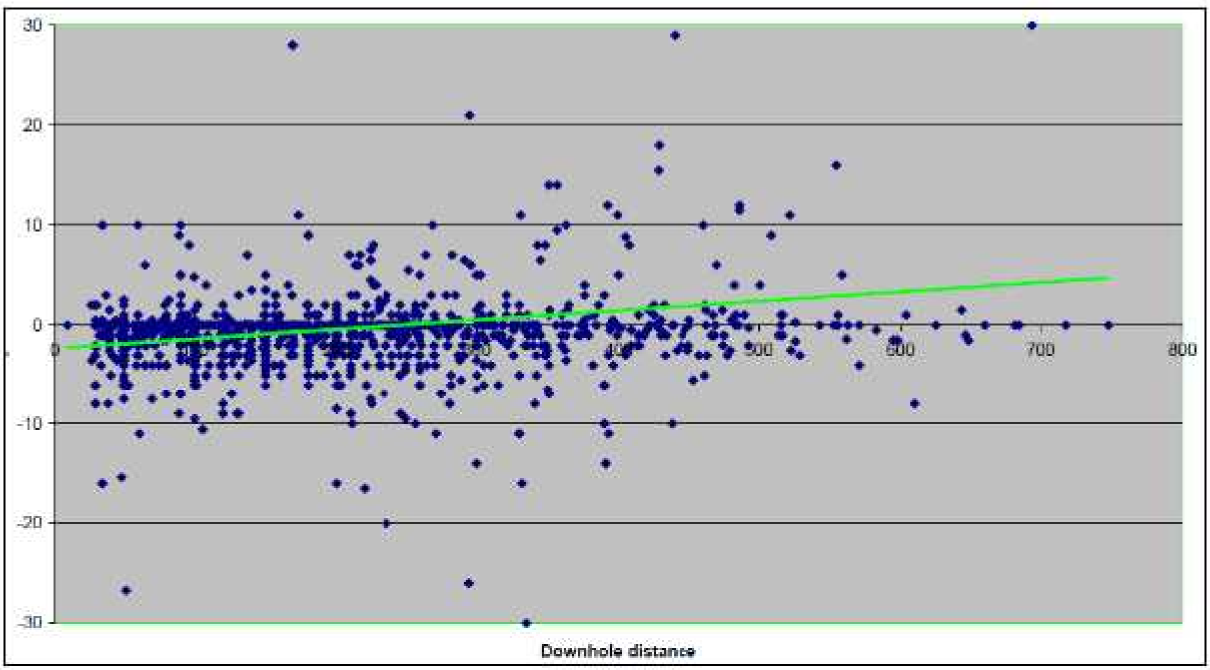


Figure 10-3: Changes in Azimuth with Depth for drilling prior to 2011

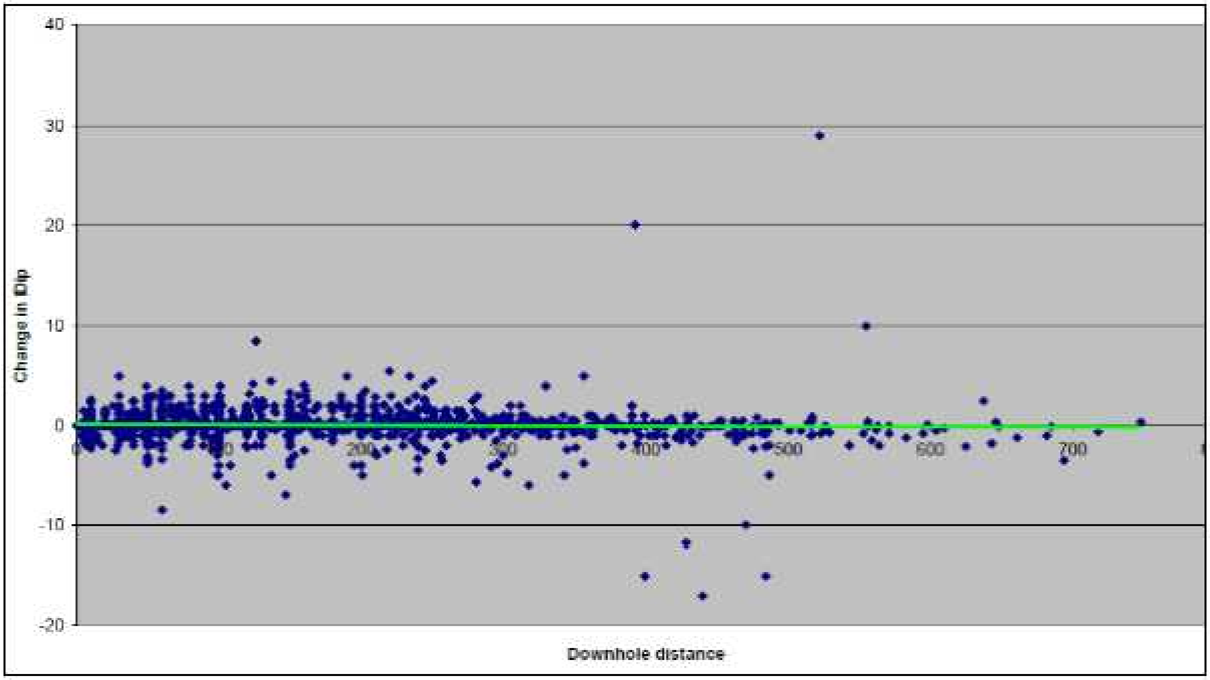


Figure 10-4: Changes in Dip with Depth for drilling prior to 2011.

Several corrections in the acQuire database were made by Newmarket Gold for typographic errors of the single shot surveys. Also, surveys not present in the database were added from source data (MC005 and MC006). Gyro surveys of MC002, MC005 and MC006 were also reprocessed, due to an incorrect application of the grid rotation.

10.2.2 Core recovery

Core recovery data is available for nine diamond holes, all of which were drilled by Hill 50 between 1997 and 2001 (Table 10-5). Overall the recorded recoveries are considered to be acceptable.

Table 10-5: Core recovery of drillholes prior to 2011

Hole ID	Average recovery%
MD045W1	100
MD045W2	97
MD053	100
MD053W1	98
MD055	98
MD055W1	97
MD055W2	96
MD055W3	98
MD056	99

Verification work completed by Newmarket Gold identified several holes which had core recovery recorded (MD017 - 19) quantitatively as 'Good'. There are some recovery records in the Database which conflict with measurements in the source data (MD014, MD024, MD029 and MD30), and geotechnical logs.

11 Sample Preparation, Analysis, and Security

11.1 Sampling Techniques

Samples used to inform the Maud Creek block model estimate are sourced from both diamond drill core and reverse circulation chip samples collected over the last 20 years.

11.1.1 Reverse Circulation Sampling for the 2011 drilling program

Figure 11-1 outlines the sampling procedure for the 2011 drilling campaign. Sample intervals (1.0 m) are washed and sieved by a geologist and then inspected to determine its geological attributes. Geology is entered directly onto standard logging sheets in either hard copy or digital form via a portable computer using standardised geological codes. Each washed sample is then stored in a chip tray and stored in a shed at the core farm for future reference.

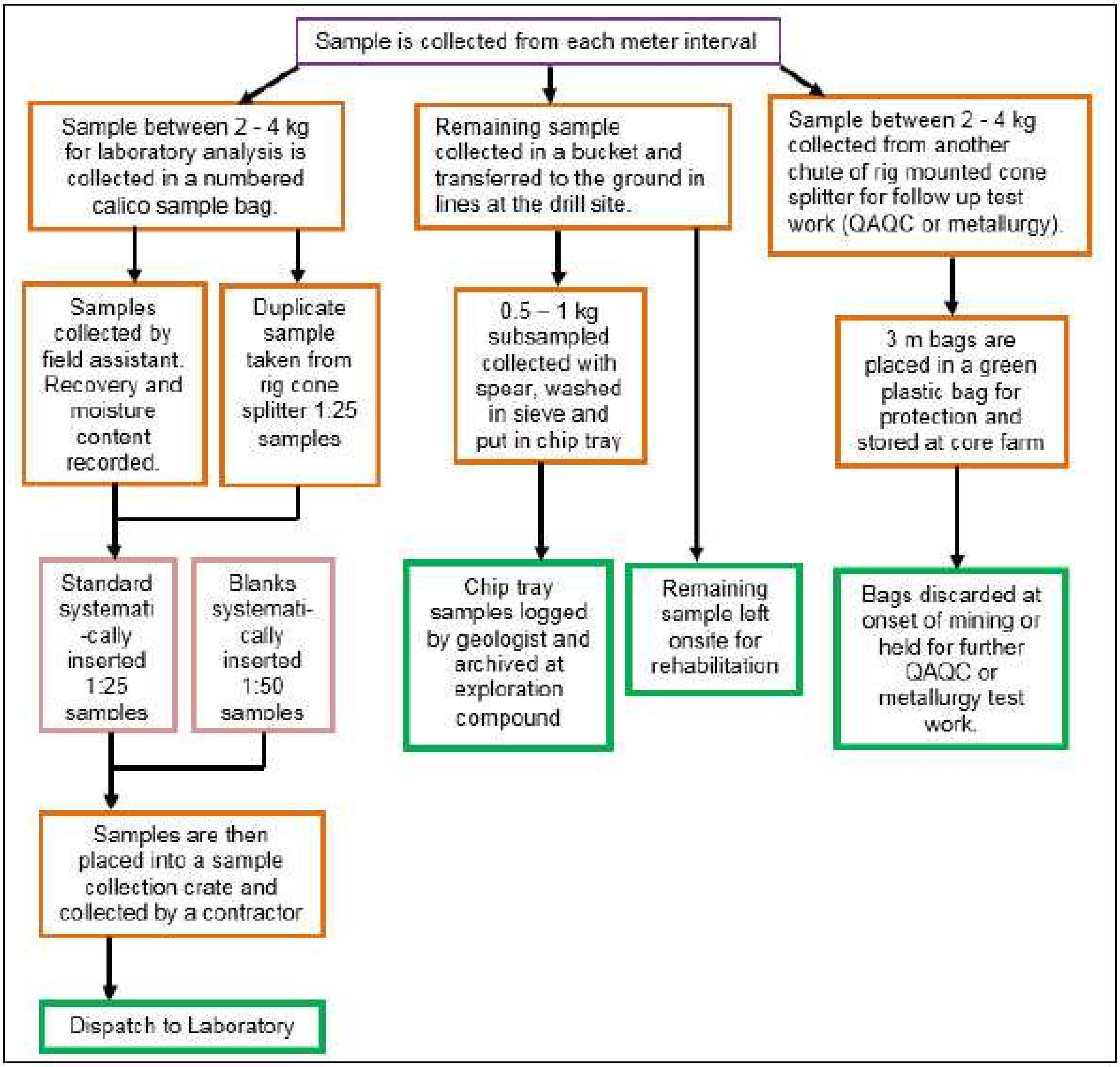


Figure 11-1: Reverse Circulation Sampling flow chart

11.1.2 Diamond Sampling for the 2011 drilling program

Much of the drill core produced from the Maud Creek area is composed of barren sediments and tuff. Therefore, not all diamond drill core is required to be sampled.

Core orientations were marked on the bottom of the core using a Camteq Orishot tool at the end of each rod. Drill core was then orientated by technicians based on these orientation marks. The Geologists then log each hole for weathering, lithology, structures, alteration, mineralization and geotechnical information. Zones of core loss are identified and marked by inserting marker blocks recording the exact length of the core loss.

At the completion of logging geologists mark the core for sampling and photograph each tray dry and wet. Samples intervals are chosen based on lithological or mineralization contacts. Sample boundaries are often made at pre-existing breaks; otherwise the half core is cut perpendicular to the core axis using an Almonte automated diamond core saw.

Minimum sample size is 0.3 m and maximum size is 1.5 m. The core was cut so as to divide the mineralization in half whilst preserving the orientation line. Some drillholes were sampled for their entire length and some were sampled from 20 – 50 m in the hanging wall through to the end of the hole.

11.1.3 Sampling prior to 2011

Details of the sample collection, preparation and quality control techniques employed by each of the previous operators of Maud Creek are not fully documented. Procedures documented in annual reports written by Kalmet Resources (1996) note that:

Sampling techniques varied for each drilling program;

A review of the assay database, analytical quality control and sampling techniques was undertaken in July 1996 by Geocraft Pty Ltd, an independent consultant;

Two metre composite RC samples were collected by riffle splitting;

5-6 kg samples were dispatched to Alice Springs, where they were riffle split to a nominal 3kg prior to pulverizing of the entire sample;

11.2 Data Sampling and Distribution

The Maud Creek model has been shown during validation to be subject to varying drillhole density and sample locations, which has affected lode geometry. Within the upper/central parts of lodes the drilling is regular and of sufficient density, but subject to decreasing densities and irregular spacing at depth.

11.3 Testing Laboratories

11.3.1 2011 Drilling program

Assaying of the drill core and reverse circulation samples was completed by either NAL at Pine Creek, the NTEL or the ALS labs in Darwin. All laboratories used are independent of Newmarket Gold and are well known to SRK Consulting as competent assayers. Once the assaying laboratory' s personnel receive the drill or chip samples they undertake sample preparation and chemical analysis. Results are returned to Newmarket Gold staff, which validate and input the data into relevant databases.

All analytical work including sample preparation, analytical procedures, QA/QC measures and associated security and chain of custody procedures have been completed in accordance with the established protocols routinely used by Newmarket Gold.

SRK Consulting considers that these procedures and protocols are of acceptable quality and are broadly consistent with international best practice standards. Lab visits have been conducted by Newmarket Gold staff to meet with the management of the laboratories and to inspect the facilities.

11.3.2 Sampling prior to 2011

Various reports written by previous tenement holders document a range of laboratory testing or investigations. The 2006 Snowden report summarised these assessments (Table 11-1):

Table 11-1: Summary of QAQC reports written for Maud Creek

Company	Report	Year	Description
Placer	Report NT25/91	1991	202 Check-analysis by ALS Townsville of samples previously analyzed by Classic Laboratories
Kalmet	MG03/002	June 1994	Comparison of gold results in 3 sets of 'twinned' RC and Diamond Drillholes
Kalmet	KMT196 Table 4	1995	Resampling of 18 intervals in 11 holes that reported > 20 g/t Au
Kalmet	KMT196 Section 5.2.3	1996	Rectification of deliberate sample corruption in Laboratory (1297 samples)

11.4 Sample Preparation

11.4.1 2011 Drilling program

The following sample preparation activities (Figure 11-2) were undertaken by Newmarket Gold staff for the 2011 drilling program:

Standards, blanks, barren quartz flush and duplicates placed in pre numbered calico bags;
Sample is placed into calico bags;
Calico bags loaded into green plastic bags with the sequence of samples in the bag labelled on the outside;
The green plastic bags were then placed into dispatch cages to be picked up by courier and taken to the Laboratory.
At the completion of each hole the core trays are stored in racks for future retrieval.

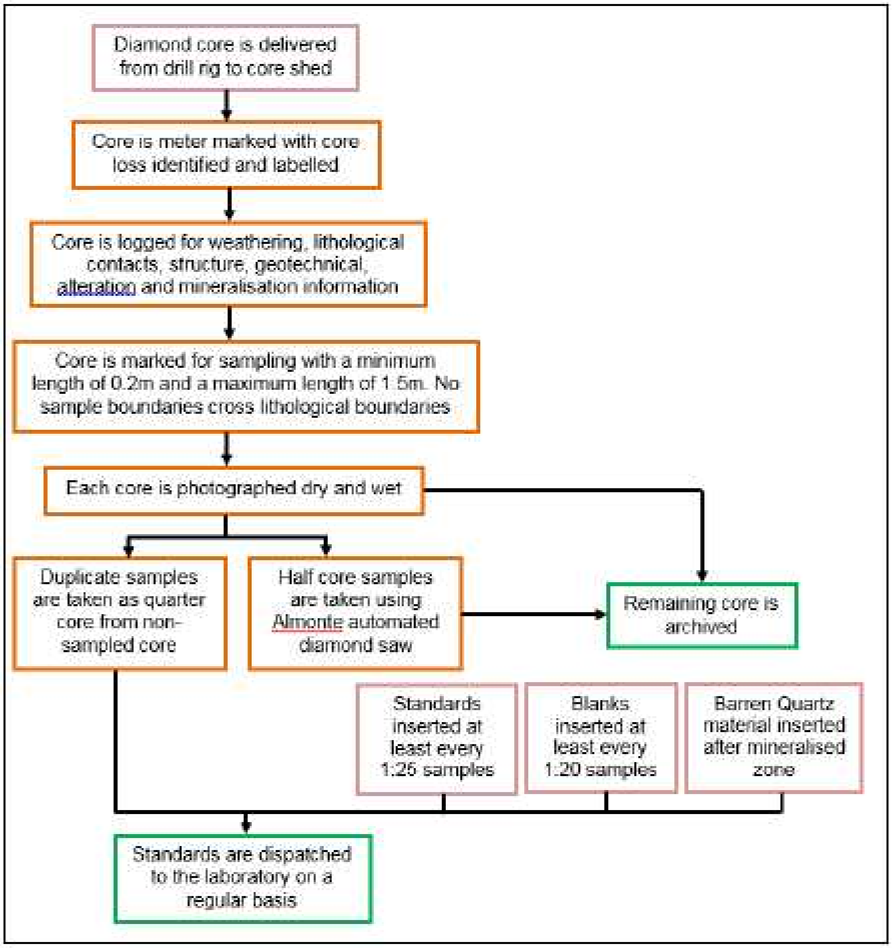


Figure 11-2: Diamond Drilling Sampling flow chart

The primary commercial laboratory used for the Maud Creek drilling campaign was Northern Territory Environmental Laboratories (NTEL), (now Genalysis), with Australian Laboratory Services (ALS) in Darwin acting as an umpire lab. Samples sent to ALS were prepared in Darwin and then sent to either the ALS laboratory facilities in Perth or Townsville for analysis.

The following sample preparation activities are undertaken by laboratory staff (Figure 11-3):

- Samples are received and checked against the submission sheet;
- Average sample weight for the submission is taken;
- Each sample is then dried at 105°C until fully dry;
- Entire sample is initially crushed in a jaw crusher to approximately 2mm;
- Each sample is then rotary split with 300g taken for milling and assay and remainder set aside as a coarse reject and returned to Newmarket Gold;
- The 300g sample is then milled to pass through a roll crusher to 2mm;
- Samples are riffle split into two sub-samples – one is milled while the other is retained as a coarse reject and returned to Newmarket Gold;
- The retained sub-sample is milled to 85% passing 75µm with 1 in 20 samples wet screened to check for compliance;
- Each milled pulp samples is further split to provide 25g for fire assay (FA25) with an AAS finish and <1g used for multi-element, if required. At ALS 30g of the pulp is weighed off for fire assay with an AAS finish (AA26).
- Any remained pulp sample is kept for future analysis and returned to Newmarket Gold.

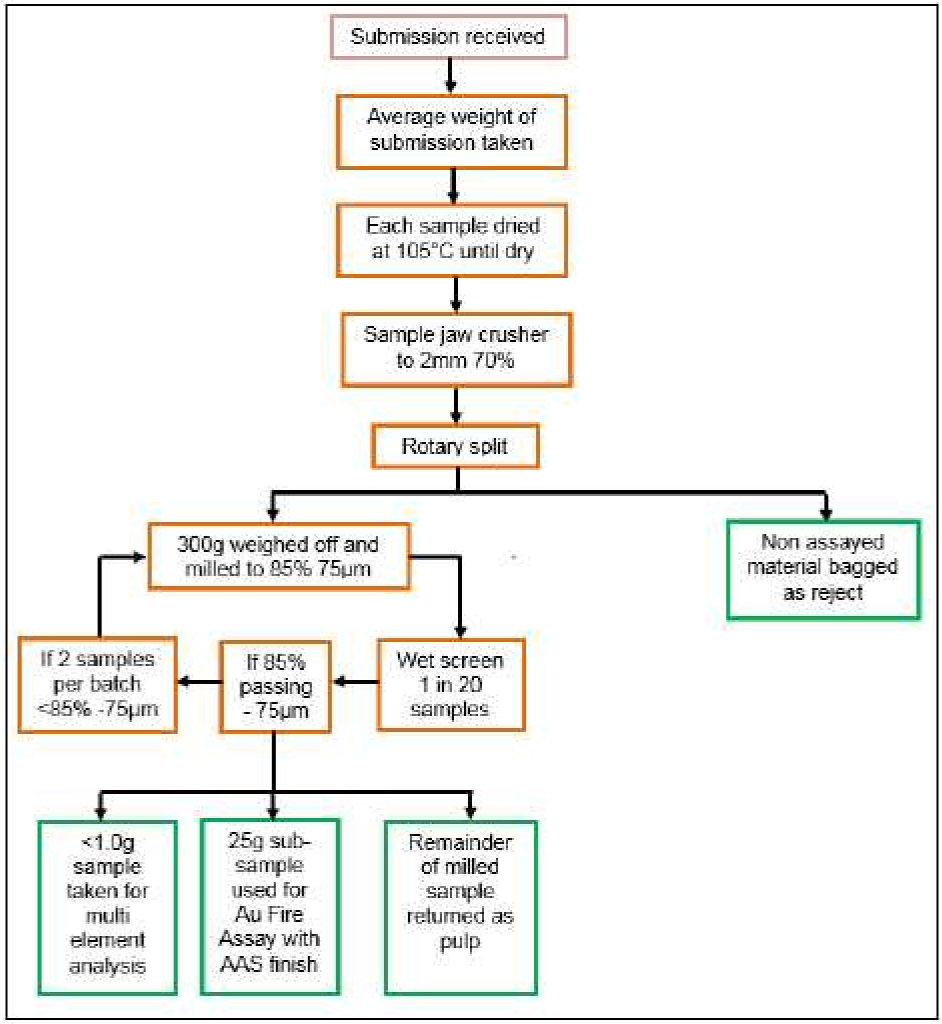


Figure 11-3: Laboratory sampling flow chart

11.4.2 Sampling prior to 2011

Multiple owners have generated the Maud Creek data set in several phases over the last 20 years. The following references were noted by Snowden in 2006:

Kalmet Resources engaged Lantana Exploration Pty Ltd in 1995 to re-enter all the analytical data obtained up to that time. This work included all drillhole data up to and including hole number MCP061;

Kalmet 1996; engaged geological consultant Geocraft Pty Ltd to verify the database and analytical procedures;

Kilkenny 1998; the MRT 1998 resource study states that data files were merged into an Access database and validated using MRT internal systems and no significant errors were detected;

Harmony 2003; a competent person has classified the Mineral Resource estimates for the Maud Creek Project in accordance with the JORC Code and these estimates have been released by Harmony to various Stock Exchanges. This chain of Competent Person Statements as required under the JORC Code has been relied upon by Terra Gold to attest to the validity of the drilling data.

Terra Gold has undertaken a validation of the database using Micromine validation routines and reported that no significant errors were detected.

11.5 Sample Analysis

11.5.1 2011 Drilling program

Maud Creek drill core and RC samples were assayed for gold and multi element analysis. Gold grades are determined by fire assay/ atomic absorption spectroscopy (AAS) and multi element (silver, arsenic, bismuth, calcium, cobalt, chromium, copper, iron, potassium, manganese, molybdenum, sodium, nickel, lead, antimony, tin, titanium, zinc, zirconium) by ICP Atomic Emission Spectrometry. The following procedure is undertaken:

- 50 g of pulp is fused with 180 g of flux (silver);
- Slag is removed from the lead button and cupellation is used to produce a gold/ silver prill;
- 0.6 mL of 50% nitric acid is added to a test tube containing prill, and the test tube is placed in a boiling water bath (100°C) until fumes cease and silver appears to be completely dissolved;
- 1.4 mL of hydrochloric acid (HCl) is added;
- On complete dissolution of gold, 8 mL of water is added once the solution is cooled; and
- Once the solids have settled, the gold content is determined by fire assay/atomic absorption AAS.

The following procedure is undertaken for multi element analysis:

- A 0.25g sample is pre- digested for 10-15 minutes in a mixture of nitric and perchloric acids;
- Hydrofluoric acid is added and the mixture is evaporated to dense fumes of perchloric;
- Residue is leached in a mixture of nitric and hydrochloric acids;
- Solution is then cooled and diluted to a final volume of 12.5mls;
- Elemental concentrations are measured simultaneously by ICP Atomic Emission Spectrometry.

11.5.2 Sampling prior to 2011

- Kalmet utilised ALS in Alice Springs for gold and arsenic assays;
- All samples were analyzed by ALS in Alice Springs for gold by fire assay method PM209 and for arsenic by AAS method G003 or G102;
- Selected holes as MD21 to MD31 were assayed for copper, lead, zinc, silver, nickel, antimony, bismuth, chromium by AAS method G102. Samples from MD21 were also analyzed for Hg by AAS method G008;
- Check samples for gold were analyzed by Analabs in Townsville;
- Duplicate check samples were assayed for gold by either ALS in Alice Springs or Assay Corp in Pine Creek;
- SG measurements were completed on 22 oxide and transition mineralization and wall rocks, selected from MD15 to MD19. Measurements were completed by Assay Corp in Pine Creek.

11.6 Laboratory Reviews

The two laboratories used by Newmarket Gold for Maud Creek offer different preparation techniques with the 25 g fire assay by NTEL and a 30 g fire assay by ALS. The following summarises findings with respect to assay work from the two independent laboratories:

- There are some errors in the datasets. Typographical errors, wrong standards, recorded/sent to the lab, obvious swaps in the databases. These errors are collaborative from both the laboratories and the database operator;

ALS report lab standards while NTEL do not;
Outright errors should not be appearing in the database (standards and replicates);
Proper control charting methods should be applied to fire assay batches that indicate standards outside proper control limits;
The lack of blanks inserted prior to sample submission needs to be addressed;
It is of the opinion of SRK that the sampling preparation, analysis and security procedures are all adequate for use in these Mineral Resource and reserve estimates.

NTEL is an independent laboratory based in Darwin. The relationship between NTEL and Newmarket Gold is on a client/ supplier arrangement with a contract in place for service. ALS laboratories are certified using the ISO9001:2008 accreditation (Quality Management Systems - Requirements). They also hold the NATA Technical accreditation under ISO17025:2005.

11.7 Assay Quality Assurance and Quality Control

11.7.1 Standard Reference Material

2011 Newmarket Gold drilling program:

Certified standards are submitted to the laboratory on a regular basis. A standard is inserted into every batch every 117 samples during the RC program and every 26 samples during the diamond drilling program (or less).

There were initially 3 standards across all ranges used during the RC drilling program and 5 standards across all ranges used during the diamond drilling program (Table 11-2).

Each standard for each drill type is charted chronologically to check for compliance and any progressive trends, which may be apparent. An example of the chart used to chronologically check the standards is presented in Figure 11-4.

A total of 48 standards were used against the 1243 samples taken for the diamond drilling program with 6 standards inserted for the 699 samples taken for the RC program. No laboratory standards were inserted.

Table 11-2: Standard ST202/5355 Compliance table for 2011 drilling program

Standard	ST202/5355
Recommended value	2.37
Mean Result	2.35
AUD% difference versus RV	-1.0
Standard Deviation	0.07
Number of assays	17
Number > -2SD	0
Number > +2SD	0
% +/- 2SD	100

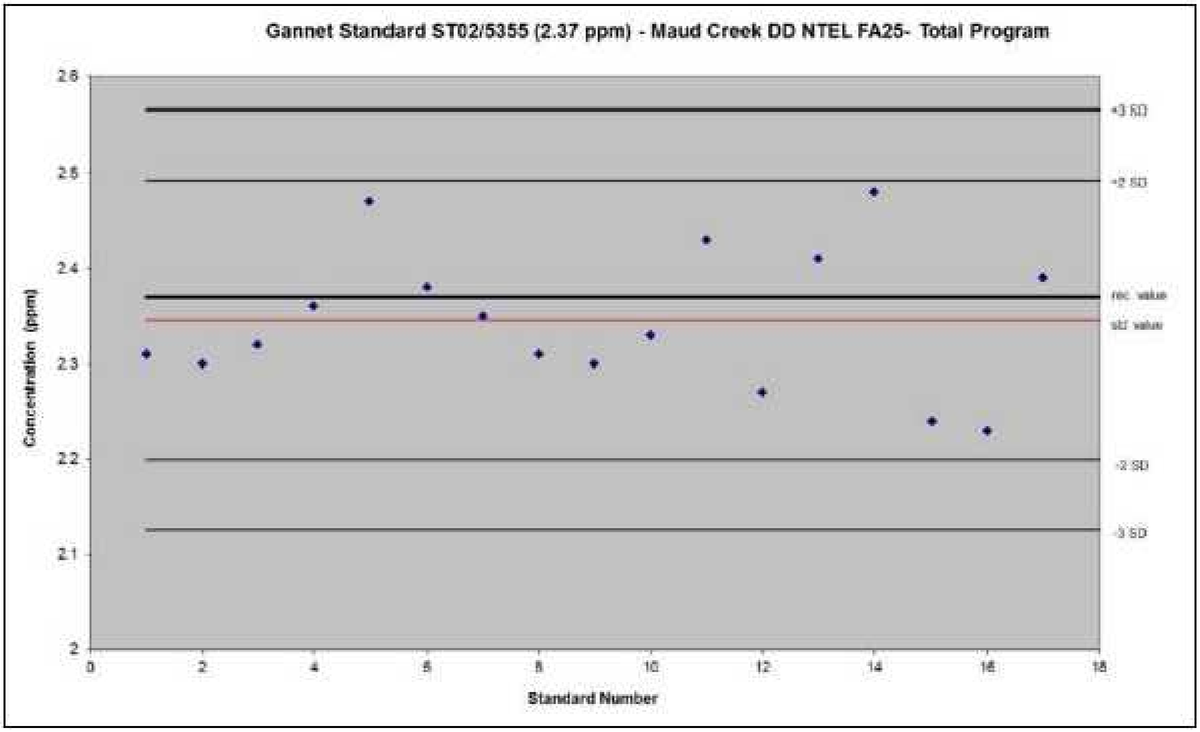


Figure 11-4: Standard ST202/5355 Compliance chart for 2011 drilling program

Sampling prior to 2011

The Maud Creek digital database notes that four certified standards were inserted during the Maud Creek drilling program run by Placer between 1990 and 1991 (Table 11-3).

Table 11-3: Maud Creek Certified Laboratory Standards

Standard	Grade (g/t Au)
OxE20	0.548
OxG22	1.035
OxH19	1.344
OxH29	1.298

Figure 11-5 to Figure 11-8 are plots sourced from pre 2011 drill data, and show difference (%) and absolute difference (%) for each standard assay on the left-hand vertical axis, with gold grade (g/t) on the right-hand vertical axis. The standard value is plotted in purple, and the CRM lab value is plotted in blue. Values of greater than 20% difference between certified value and the determined value appear to be related to submission error or incorrect standard.

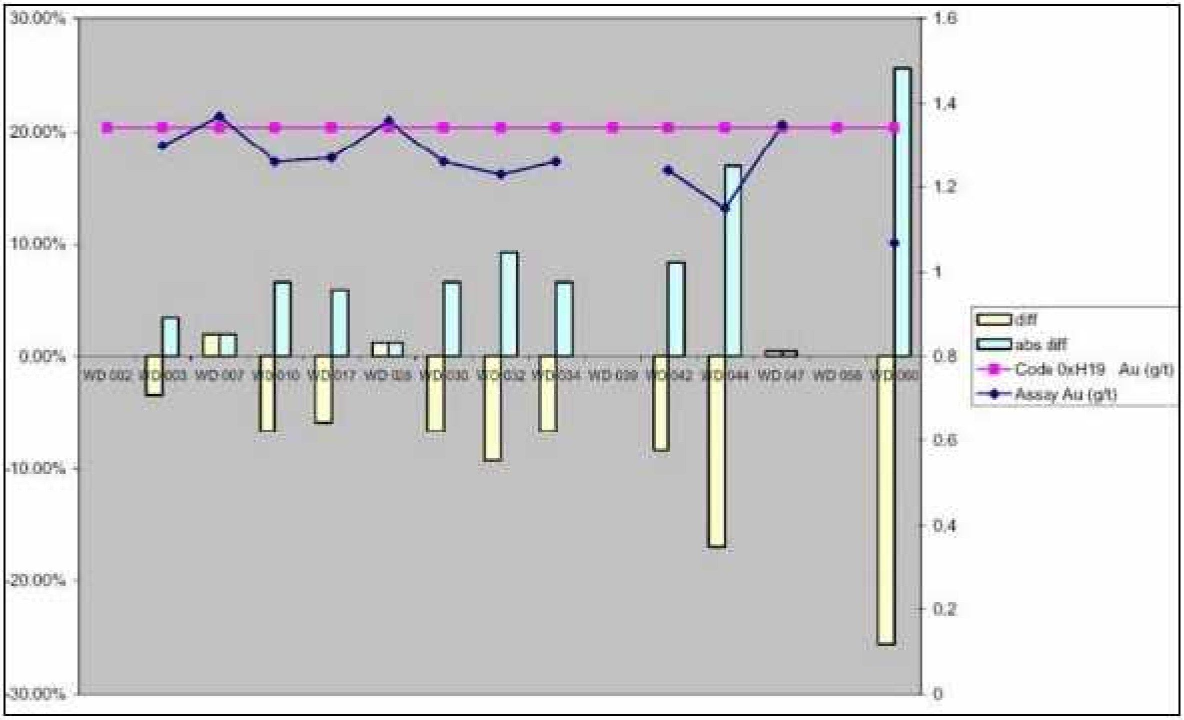


Figure 11-5: Standard OxH19

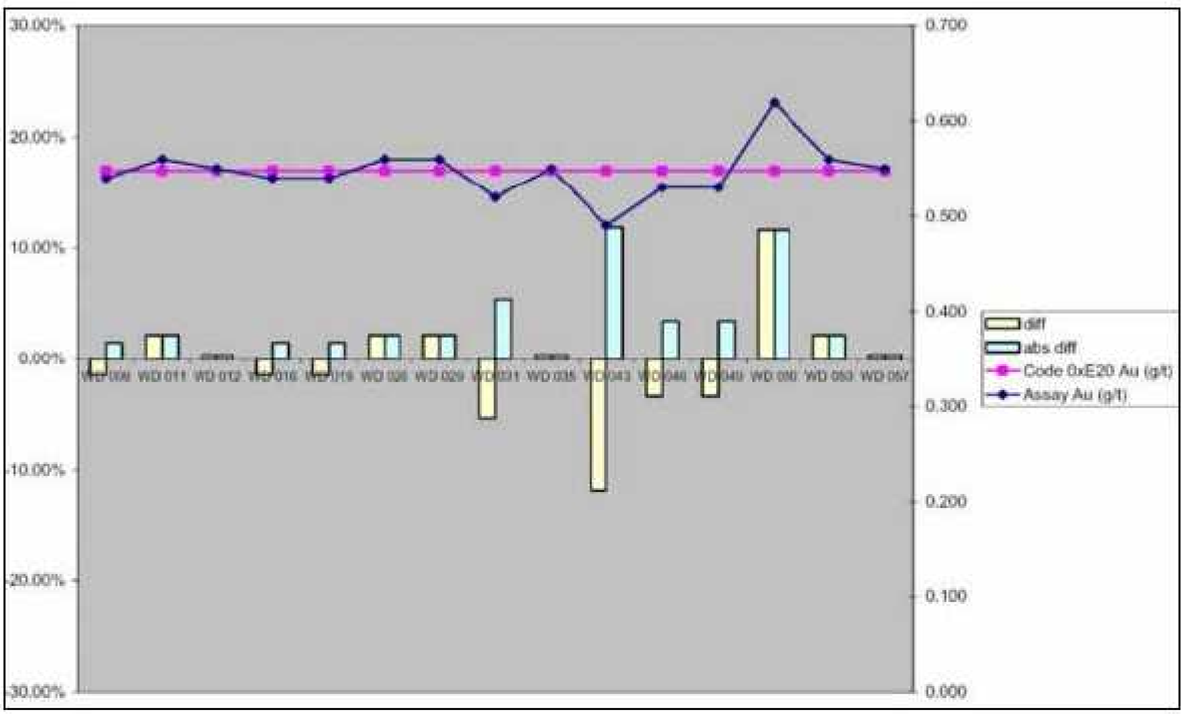


Figure 11-6: Standard OxE20

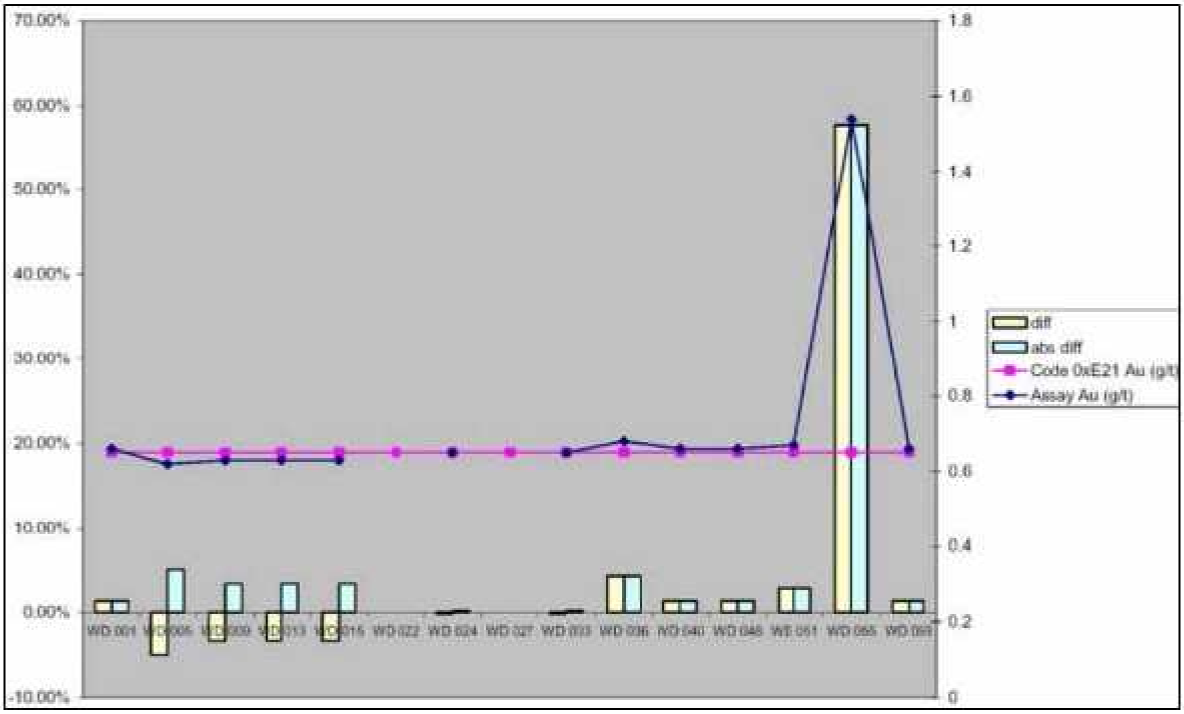


Figure 11-7: Standard OxE21

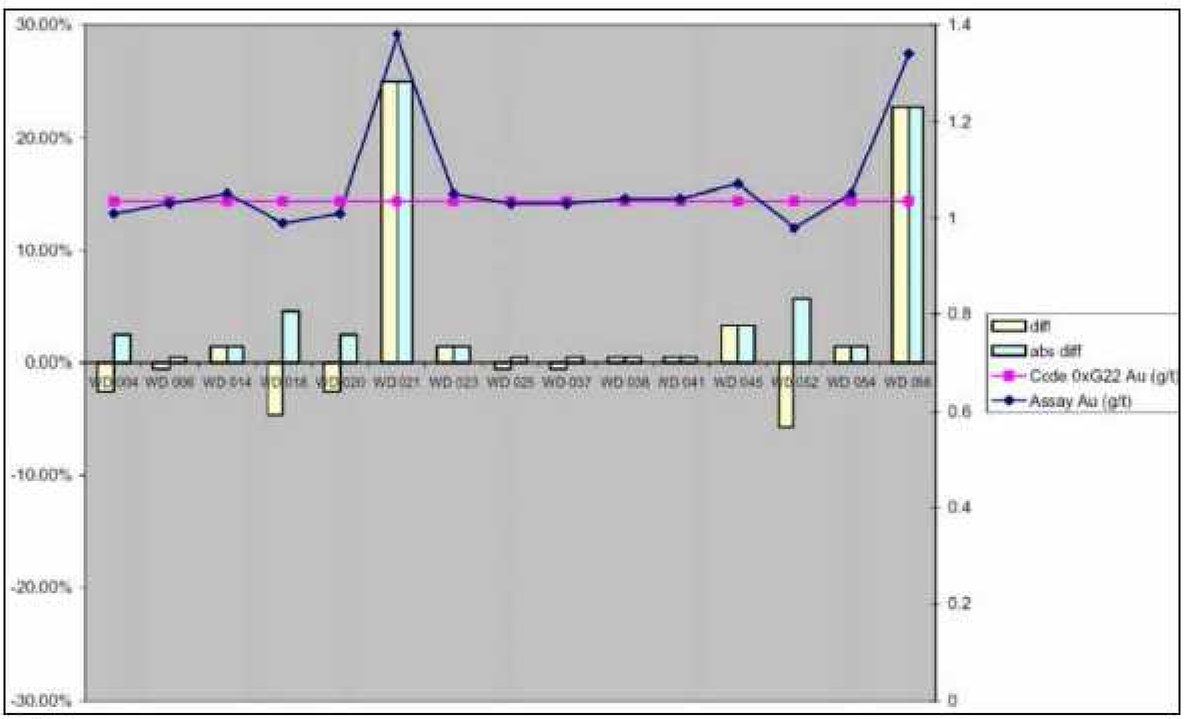


Figure 11-8: Standard OxG22

11.7.2 Blank Material

2011 Newmarket Gold drilling program:

Blank materials included in the sample stream were derived from several sources; barren core, barren coarse rejects, crushed Bunbury Basalt (from Gannet Holding Pty Ltd, referred to in this report as blank). Blank results above 0.02 g/t Au are queried and any issues resolved. Results are chronologically charted to visually check compliance (Figure 11-9). No blanks were inserted into the Maud Creek RC drilling program. For the diamond drilling program, a total of 72 blanks were inserted with 98.6% at or below 0.02 g/t Au.

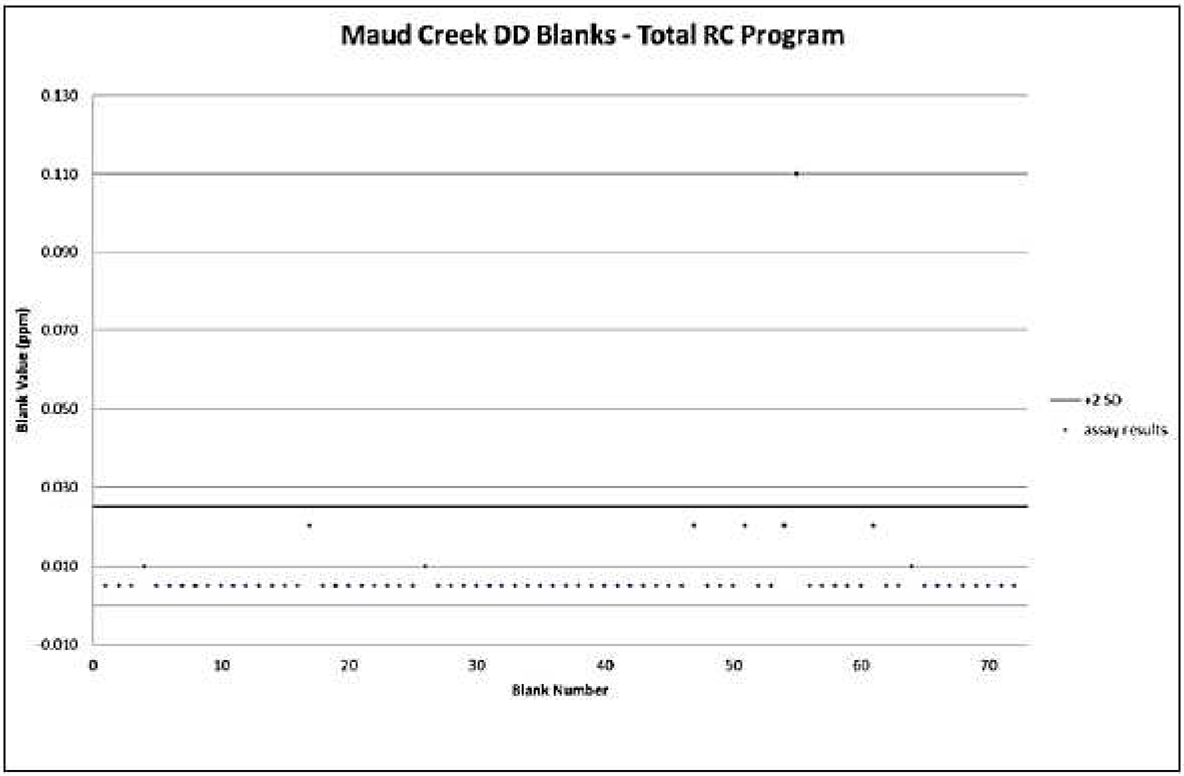


Figure 11-9: Compliance chart for blanks used in the 2011 Maud Creek drilling program.

Sampling prior to 2011

There was no evidence in any reporting indicating the insertion of blank material with samples submitted to laboratories.

11.7.3 Duplicate Assay Statistics

2011 Newmarket Gold drilling program:

Relative precisions have been used to analyze the precision of duplicate samples. The relative precision is a measure of dissimilarity, that is, if both distributions are exactly the same, this value will equal zero increases as the distributions become more dissimilar.

In this report, relative precision has been calculated using all data pairs for the ranges of below detection (<0.01 g/t) to 0.20 g/t Au, 0.21 to 0.5 g/t Au, 0.51 to 0.7 g/t Au, 0.71 to 1.00 g/t Au, 1.01 to 1.40 g/t Au, 1.41 to 5.00 g/t Au and >5.00 g/t Au. This is to isolate the large conditional variance of errors associated with assay determinations near both lower and upper analytical detection limits and to selectively analyze results within these set ranges.

An example of the analysis tables for the 2011 Maud Creek drill program is given in Table 11-4 to Table 11-6 and Figure 11-10.

Table 11-4: Duplicate analysis table for the 2011 drilling program

Repeat	Maud Creek DD NTEL FA25 Total Program
Mean original results:	0.06
Mean repeat results:	0.07
Number of assays:	39
Standard Deviation:	0.07
Sum of Differences:	-0.39
Sum of Diff * Diff:	0.18
Mean Difference:	-0.01
% Results within +/- 2SD	102
Results within 30% precision level	86
Average absolute% Difference:	24
% Assays original <or = repeat	83

Table 11-5: Duplicate correlation table for the 2011 drilling program

Range (g/t)	Original vs Repeat
Combined	0.952
<0.20	0.777
0.21 - 0.50	-
0.51 - 0.70	-
0.71 - 1.00	1.000
1.01 - 1.40	-
1.41 - 5.00	-
>5.01	-

Table 11-6: Duplication R Table for the 2011 drilling program

Range (g/t)	# of assays	% of total #	Mean original	Mean repeat	% diff between means (bias)	Average% diff between assays (bias)	Absolute average% diff between assays (total error)	Standard Deviation
<0.20	38	97	0.03	0.04	-21.8	-9.00	24	0.07
0.21 - 0.50	0	0	-	-	-	-	-	-
0.51 - 0.70	0	0	-	-	-	-	-	-
0.71 - 1.00	1	3	1.08	1.19	-10.2	-10.19	10	0.00
1.01 - 1.40	0	0	-	-	-	-	-	-
1.41 - 5.00	0	0	-	-	-	-	-	-
>5.01	0	0	-	-	-	-	-	-
Total	**39**	**100**	**0.06**	**0.07**	**-16.5**	**-1.3**	**24**	**0.07**

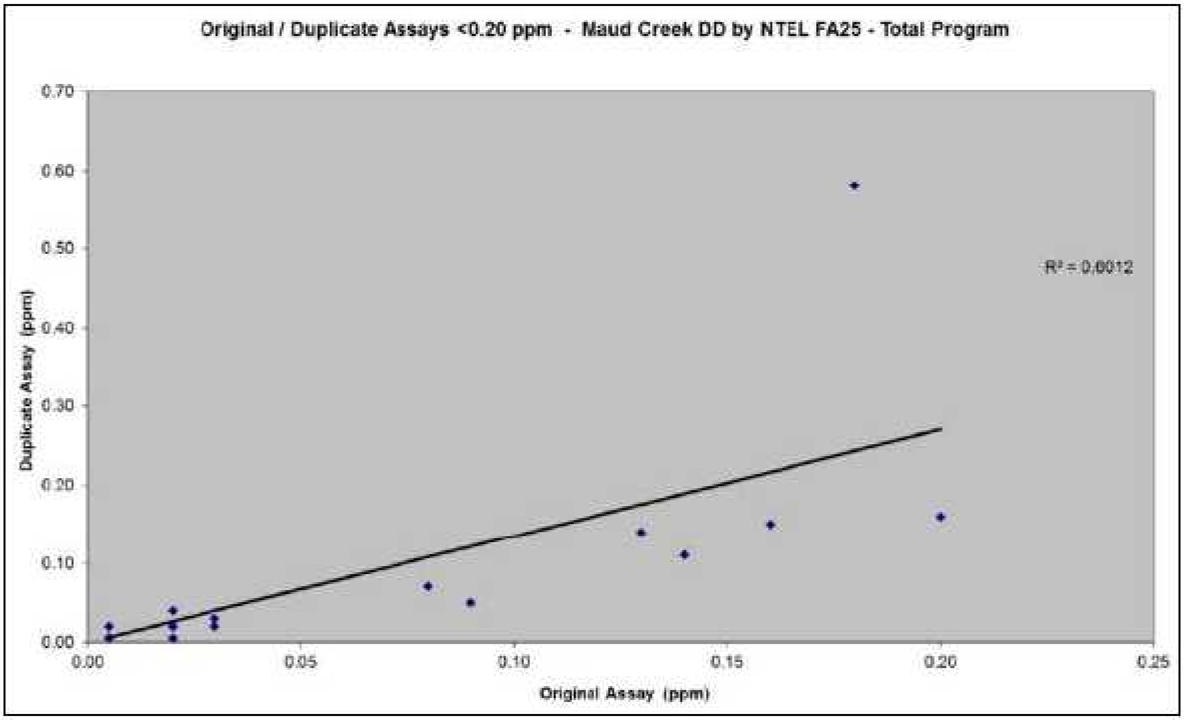


Figure 11-10: Duplicate correlation plot; range <0.2 g/t for 2011 drilling program

Eighty Six per cent of diamond duplicates for Maud Creek fall within the 30% precision level. All but one of the 39 original samples is above 0.2 g/t Au with 24 original samples being below the detection limit. Of the 29 RC duplicates taken, 24 of the original samples are below the detection limit with 87% falling within the 30% precision level.

Sampling prior to 2011

The database for pre 2011 drilling contains laboratory repeat data (Au1 and Au2). Figure 11-11 is a relative difference plot of the two data sets. It would generally be expected that as the average grade of each pair of data increases the relative difference between the paired data would decrease. The plot does not indicate this, suggesting some issues with laboratory precision. This may reflect the presence of coarse or nuggety gold. It is understood that no screened fire assay analysis were undertaken to help assess whether coarse gold is an issue at Maud Creek. Furthermore it must be noted that these are (presumably) repeats initiated by the laboratory, and not blind submissions of field duplicates, which may be normally expected to show poorer precision than laboratory repeats.

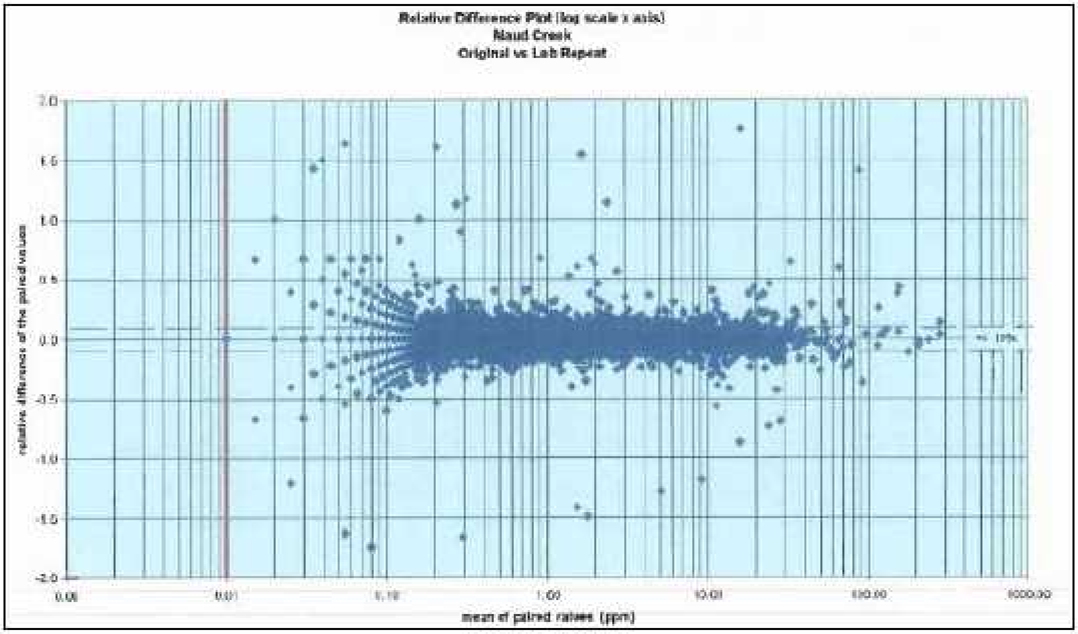


Figure 11-11: Relative difference plot original vs lab repeats for pre 2011 Maud Creek drill data

11.7.4 Internal laboratory Repeats

2011 Newmarket Gold drilling program:

Internal laboratory repeats were taken for both RC and diamond drilling at the primary laboratory (NTEL). Relative precisions have been used to analyze the precision of repeat samples. The relative precision is a measure of dissimilarity, that is, if both distributions are exactly the same, this value will equal zero increases as the distributions become more dissimilar.

In this report, relative precision has been calculated using all data pairs for the ranges of below detection (<0.01g/t) to 0.20,g/t Au,0.21 to 0.5g/t Au, 0.51 to 0.7g/t Au, 0.71 to 1.00g/t Au, 1.01 to 1.40g/t Au, 1.41 to 5.00g/t Au and >5.00g/t Au. This is to isolate the large conditional variance of errors associated with assay determinations near both lower and upper analytical detection limits and to selectively analyze results within these set ranges.

An example of the analysis tables for the 2011 Maud Creek drill program is given in Table 11-7 to Table 11-9 and Figure 11-12.

Table 11-7: Repeat analysis table for the 2011 drilling program:

Repeat	Maud Creek DD NTEL FA25 Total Program
Mean original results:	0.36
Mean repeat results:	0.35
Number of assays:	356
Standard Deviation:	0.16
Sum of Differences:	2.75
Sum of Diff * Diff:	9.55
Mean Difference:	0.01
% Results within +/- 2SD	98
Results within 30% precision level	96
Average absolute% Difference:	1
% Assays original <or = repeat	97

Table 11-8: Repeat correlation table for the 2011 drilling program

Range (g/t)	Original vs Repeat
Combined	0.991
<0.20	0.999
0.21 - 0.50	0.993
0.51 - 0.70	0.940
0.71 - 1.00	1.000
1.01 - 1.40	0.998
1.41 - 5.00	0.976
>5.01	0.998

Table 11-9: Repeat R Table for the 2011 drilling program

Range (g/t)	# of assays	% of total #	Mean original	Mean repeat	% diff between means (bias)	Average % diff between assays (bias)	Absolute average% diff between assays (total error)	Standard Deviation
<0.20	271	76	0.02	0.02	-0.5	-0.1	0	0.00
0.21 - 0.50	34	10	0.34	0.34	0.3	0.5	1	0.01
0.51 - 0.70	7	2	0.59	0.60	-2.2	-1.9	2	0.04
0.71 - 1.00	10	3	0.83	0.83	0.0	0.0	0	0.00
1.01 - 1.40	8	2	1.14	1.14	-0.2	-0.2	0	0.01
1.41 - 5.00	23	6	2.74	2.65	2.3	2.3	6	0.25
>5.01	3	1	9.13	8.87	13.5	13.5	21	2.01
Total	**356**	**100**	**0.36**	**0.35**	**2.1**	**0.1**	**1**	**0.16**

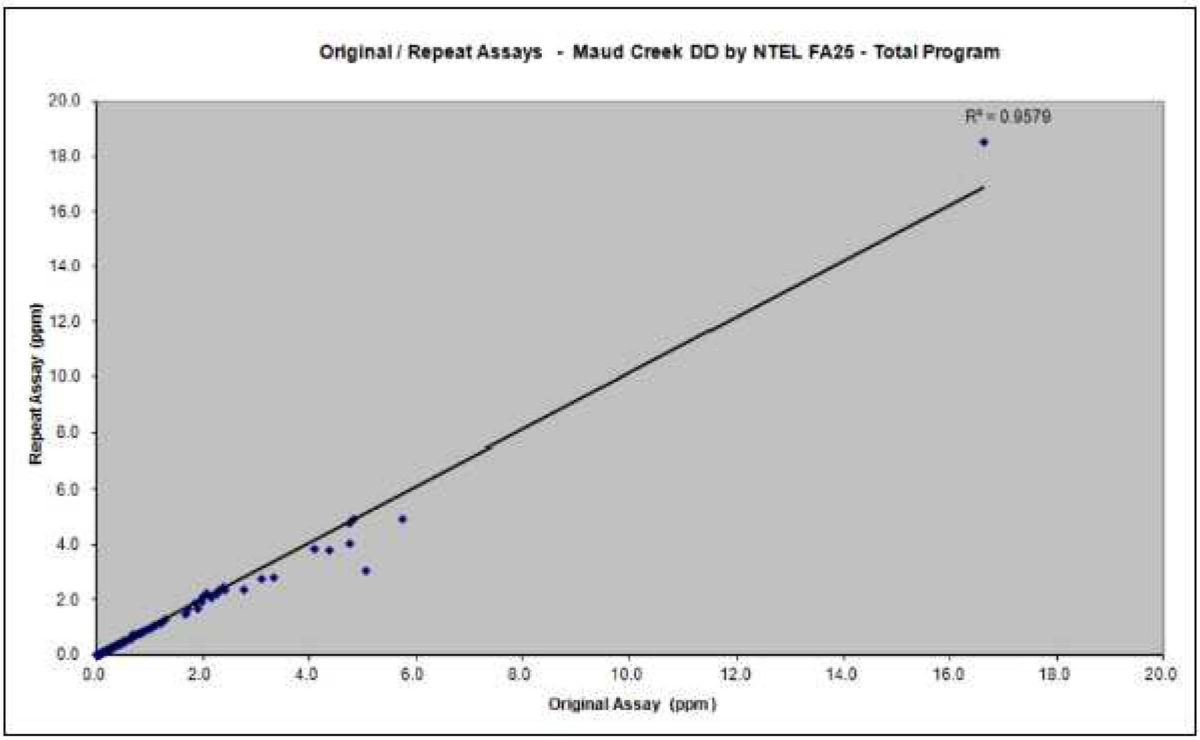


Figure 11-12: Repeat correlation plot; range <20 g/t for 2011 drilling program

Ninety six per cent of diamond repeats for Maud Creek fall within the 10% precision level. One hundred and ninety seven of the original samples are below the detection limit. Eighty five of the 356 original samples are above 0.2 g/t Au. Of the 7 RC repeats taken, none of the original samples are below the detection limit with 26% falling within the 10% precision level.

11.7.5 Inter-laboratory Repeats

2011 Newmarket Gold drilling program:

An example of the analysis tables for the 2011 Maud Creek drill program is given in Table 11-10 to Table 11-12 and Figure 11-13.

Table 11-10: Inter-laboratory repeat analysis table NTEL: ALS for the 2011 drilling program

Repeat	Maud Creek DD NTEL:ALS Total Program
Mean original results:	1.52
Mean repeat results:	1.42
Number of assays:	77
Standard Deviation:	0.35
Sum of Differences:	7.75
Sum of Diff * Diff:	9.18
Mean Difference:	0.10
% Results within +/- 2SD	95
Results within 30% precision level	73
Average absolute% Difference:	7
% Assays original <or = repeat	44

Table 11-11: Inter-laboratory repeat correlation table NTEL: ALS for the 2011 drilling program

Range (g/t)	Original vs Repeat
Combined	0.990
<0.20	0.953
0.21 - 0.50	0.964
0.51 - 0.70	0.109
0.71 - 1.00	0.743
1.01 - 1.40	0.885
1.41 - 5.00	0.907
>5.01	0.998

Table 11-12: NTEL: ALS Inter-laboratory repeat R Table for the 2011 drilling program

Range (g/t)	# of assays	% of total #	Mean original	Mean repeat	% diff between means (bias)	Average% diff between assays (bias)	Absolute average% diff between assays (total error)	Standard Deviation
<0.20	14	17	0.11	0.10	5.8	6.24	11	0.02
0.21 - 0.50	21	26	0.40	0.39	2.8	2.87	5	0.03
0.51 - 0.70	6	7	0.63	0.66	-4 .2	-4.26	7	0.05
0.71 - 1.00	7	9	0.84	0.84	0.0	0.06	4	0.05
1.01 - 1.40	6	7	1.13	1.11	1.3	1.11	4	0.06
1.41 - 5.00	24	29	2.64	2.36	10.5	7.44	10	0.61
>5.01	4	5	8.34	8.09	2.9	3.71	4	0.45
Total	**82**	**100**	**1.52**	**1.52**	**0.0**	**1.1**	**7**	**0.34**

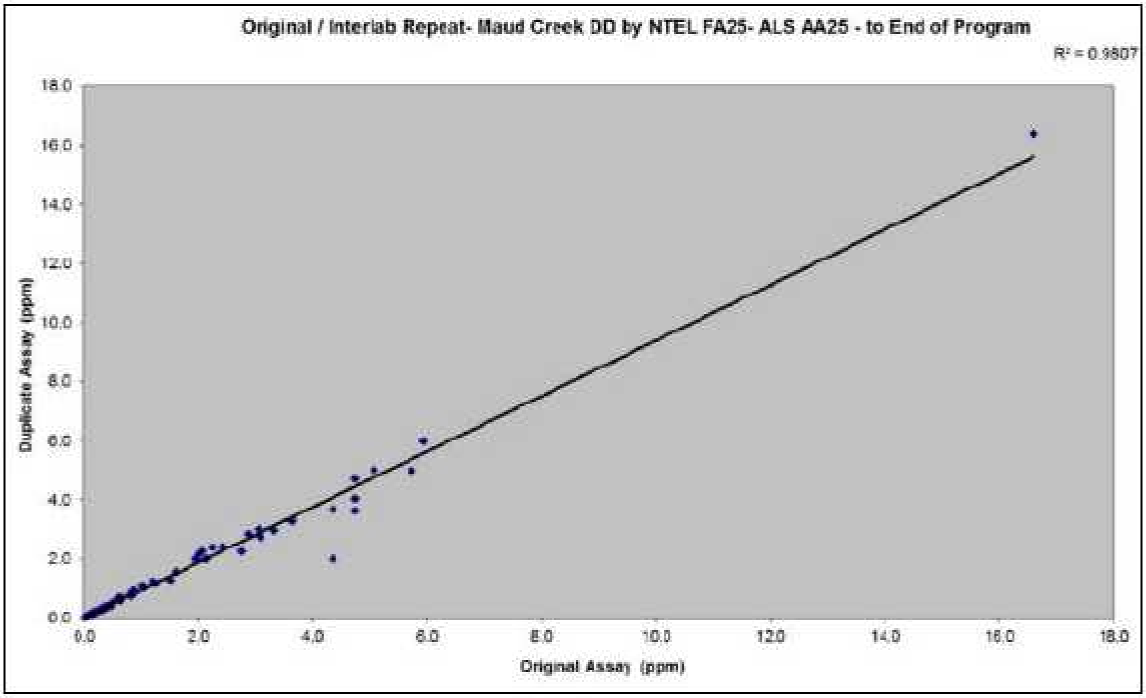


Figure 11-13: NTEL: ALS Inter-laboratory repeats for all ranges NTEL FA25: ALS AA25 for the 2011 drilling program

Seventy three percent of all diamond pulp samples fall within the 10% precision level for inter laboratory repeats. The limited population of inter laboratory repeats for both the diamond and RC programs limits this data. Seventeen percent of all RC pulp samples fall within the 10% precision level for inter laboratory repeats however only 2 samples are above 0.2 g/t with samples approaching the lower detection limit significantly affecting the precision results.

For the 2011 Newmarket Gold soil sampling program:

Soil sampling programs were undertaken at Maud Creek. 2488 soil samples were taken at Maud Creek with 82 (3.2%) of them being duplicate samples. - Samples were sent to ALS in Perth and analyzed using their Ionic Leach MEMS 23 method. Eighty four percent of duplicate samples taken from Maud Creek were within the 30% precision level.

11.7.6 Sampling prior to 2011

Some umpire analysis was undertaken between ALS (the primary laboratory) and Assay Corp laboratories. It is not known whether any common CRM was submitted to both laboratories to assist in calibrating the results. Figure 11-14 is a log Q-Q plot of the ALS data against the Assay Corp data and suggests that ALS is slightly under-reporting relative to Assay Corp.

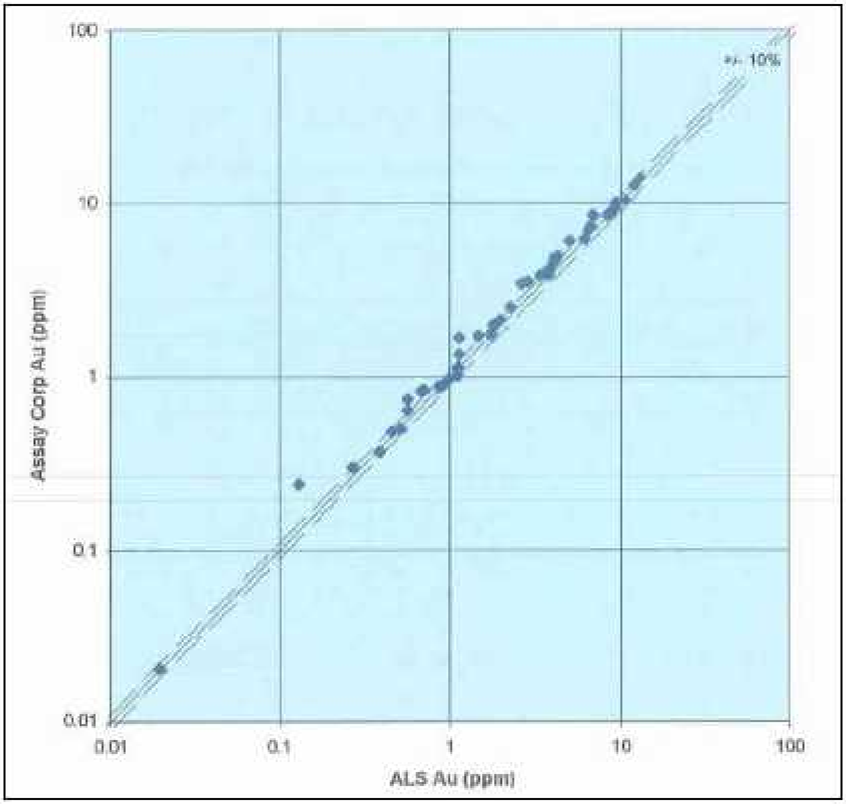


Figure 11-14: ALS: Assay Corp Inter-laboratory check analysis for pre 2011 Maud Creek drilling

Some samples were re-split, re-assayed and compared against the original data. Figure 11-15 is a log scatter plot of the re-split data which suggests a slight bias towards the original assay, although the overall correlation is acceptable. However, as the bulk of the data grades are less than 1 g/t Au, the results have little relevance to the Maud Creek estimate.

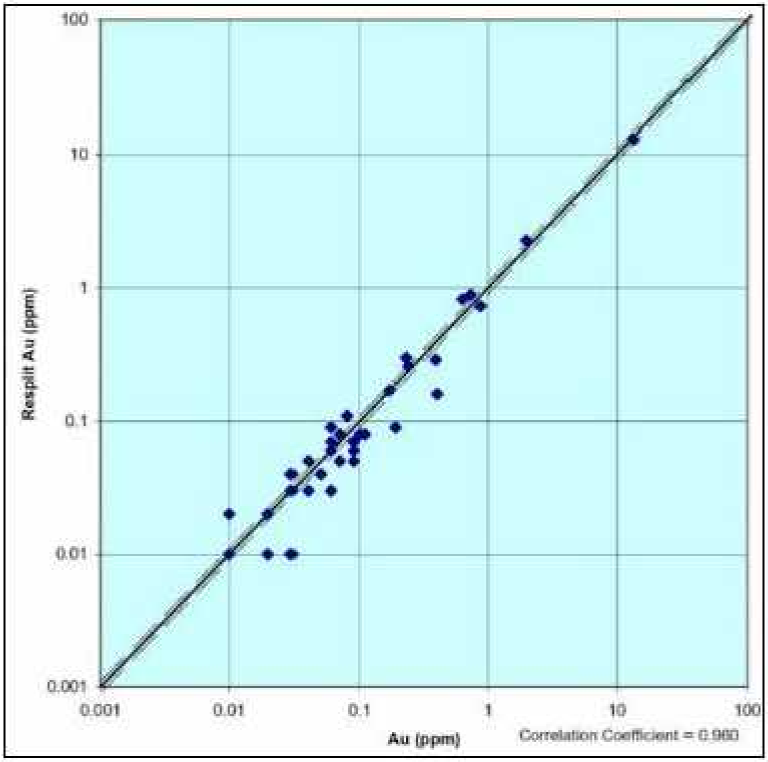


Figure 11-15: ALS Re-split check assays for pre 2011 Maud Creek drilling

11.8 Sample Transport and Security

11.8.1 2011 Newmarket Gold drilling program

A Newmarket Gold staff member is stationed on the RC drill rig while samples are being drilled and collected. At the end of shift samples were generally transported to the sample collection area where they are stored in crates as they await transportation the lab. Samples are shipped in regular intervals so they are not in crates for a length of time. These samples are located at the Brocks Creek exploration office, which can be secured if no staff member is on site.

In terms of diamond drilling, the core is collected daily from the rig and transported to the exploration office near the old Brocks Creek underground mine. Prior to the samples being transported, a photo was taken on site. This was to ensure there was a record of the material drilled before it left site, it also served the purpose of having a geological record of the drilling in case the core was damaged during transportation. The drill core is then stored in the core shed for logging and sampling. The core shed is located in a compound with security fencing. This location is locked up when no Newmarket Gold staff member is on site. Samples are cut at this location and loaded into lab crates once in calico sample bags as they await collection. These samples are then transported directly to the lab for analysis.

Once assaying is complete the results are returned in digital format to the data entry personnel employed by Newmarket Gold. These files are then loaded directly into a Datashed database. Validation via a visual comparison of standard and blanks against received values. Any questionable results are then raised with the laboratory and resolved. Submissions outside given QAQC guidelines are rejected and not loaded until resolved by the laboratory. The Datashed database is located at the Exploration office and the software is a SQL database with built-in security limiting access to people outside the Company network.

11.8.2 Sampling prior to 2011

After taking custody of the drill core, Geologists' conducted an industry compliant program of geological logging, photography, density measurements, and core sampling. Core was logged in detail onto paper and then entered into the project database. A site visit was completed in January 2006 by Snowden and the drill core was found to be well handled and maintained.

11.9 Conclusions

11.9.1 2011 Newmarket Gold drilling program

The results from the QAQC analysis of drilling has indicated a good level of confidence in assay grades for use in the resource model. The following recommendations for improvements in the current procedures are:

- An immediate follow up with the laboratory when controls fail;
- Increase in the regularity of blank material within the sample stream with 1:50 for RC drilling and 1:20 for diamond drilling;
- An increase in the regularity of standards inserted to the desired 1:25 rate;
- Inter laboratory repeats to meet or exceed a rate of 1:20 to original samples;
- Assay results to be thoroughly assessed for errors prior to loading;
- Conducting an analysis on barren core that is re- used to serve as blanks for future batches; and
- Regular tracking of QAQC compliance.

11.9.2 Sampling prior to 2011

Recommendations were made by Snowden in 2006 to improve the sampling procedures:

- The current QA/ QC programs should be continued for all future sample programs at Maud Creek;
- Continuation of the compilation and documentation of historical work undertaken at Maud Creek;
- Systematic analysis and reporting of the QA/QC data acquired during sampling; and
- Regular auditing of the database and sampling procedures in order to maintain the integrity of the database.

Since Newmarket Gold has taken ownership of Maud Creek, the recommendations made by Snowden in 2006 have been incorporated into sampling procedures.

12 Data Verification

A thorough examination of all available information was conducted by SRK and Lee Beer of Newmarket Gold. A summary of the issues identified can be found below:

- Several elements from historical assaying had not been imported into the database, notably sulphur and arsenic
- Differences in end of hole depths between the Datashed and acquire databases
- Grid conversion issues between regional MGA and calculated local grid coordinates
- Lack of clarity when distinguishing between original and re- drilled holes (same collar coordinates but differing end of hole depths)
- Issues with gyroscopic surveys due to the magnetic correction being erroneously applied.
- Missing assays from the Datashed database
- Missing assays from original source (pdf document)
- Different assay values between Datashed and acQuire databases
- 'Self-referencing' field duplicates
- Conflicting core recovery values between Datashed and acQuire databases
- Overlapping intervals in geology logging
- Interval gaps in geology logging
- Unrecognized logging codes
- Logged intervals beyond end of hole depth
- Different types of values logged for the same variable; for example, RQD logged at both a percentage and a metre value
- Missing geology logs from either digital database; only present in scanned pdf document

All issues have either been rectified by Newmarket Gold/ SRK or were deemed immaterial for the current resource estimate and will be entered/corrected in the Datashed database when time permits. SRK also conducted a site visit to verify the logging codes used in the 2011 Newmarket Gold drilling, and any available historic drilling. SRK believes the level of geological logging utilised throughout the Maud Creek drilling programs is sufficiently consistent and representative to use for a Mineral Resource Estimate.

All available QAQC reports were analyzed by SRK, to ensure the sample preparation and analysis conducted for each drill program was consistent with industry standards (Table 12-1). QAQC reports prior to 1998 did not include any analysis of the CRM's or blanks, but did include an investigation of field duplicates. Check laboratories have been utilised throughout the Maud Creek drilling programs, generally between ALS, Assay Corp and NAL. Generally QAQC reports were created by external consultants, either GEOCraft or Snowden, and concluded that the sample preparation and analysis techniques used had been appropriate and consistent with industry standards.

Table 12-1: Summary of QAQC Reports completed for Maud Creek

Company	Year	Program	Laboratory	Check Lab	CRM analysis checked	Blank analysis checked	Duplicate analysis checked	QAQC report
CGAO	2011	MC (6)	NAL	ALS	Y	Y	Y	SRK 2015/2006 Snowden
		MCRC (14)	NAL	ALS	Y	Y	Y	SRK 2015/2006 Snowden
Terra Gold	2007	TMCD (1)	SGS		Y	Y	Y	GEOCraft 2005
Terra Gold	2005	MD (2)	ALS	ALS	Y	Y	Y	GEOCraft 2005
Hill 50	2001-2002	MCP (40)	NAL		Y	Y	Y	Snowden 2005/ GEOCraft 2005
		MD (16)	NAL		Y	Y	Y	Snowden 2005/ GEOCraft 2005
Anglo Gold	1999-2000	MRC	Assay Corp	Amdel	Y		Y	AngloGold internal Standard monitoring
Kilkenny Gold	1997-1998	DRC (44)	Assay Corp		N	N	Y	Snowden 2005/ GEOCraft 2005
		GRC (17)	Assay Corp		N	N	Y	Snowden 2005/ GEOCraft 2005
		MCE (21)	Assay Corp		N	N	Y	Snowden 2005/ GEOCraft 2005
		MCW (51)	Assay Corp		N	N	Y	Snowden 2005/ GEOCraft 2005
		MD (19)	Assay Corp		N	N	Y	Snowden 2005/ GEOCraft 2005
		SRR (14)	Assay Corp		N	N	Y	Snowden 2005/ GEOCraft 2005
Kalmet	1993-1996	KR (282)	ALS	Assay Corp	N	N	Y	GEOCraft 1996/2005
		MCP (265)	ALS	ANALA BS	N	N	Y	GEOCraft 1996/2005
		MD (33)	ALS	ALS	N	N	Y	GEOCraft 1996/2005

		MRB (216)	ALS		N	N	Y	GEOCraft 1996/2005

Company	Year	Program	Laboratory	Check Lab	CRM analysis checked	Blank analysis checked	Duplicate analysis checked	QAQC report
		MRC (36)	ALS	ALS	N	N	Y	GEOCraft 1996/2005
		SWM (9)	ALS		N	N	Y	GEOCraft 1996/2005
Placer	1989-1991	WD (20)	Classic Laboratories	ALS	N	N	Y	GEOCraft 1996/2005
		WP (40)	Classic Laboratories	ALS	N	N	Y	GEOCraft 1996/2005

12.1 Site Visit

Kirsty Sheerin of SRK visited site between the 3rd and 7th August 2015 in order to examine and check the logged core that was still available at site (Table 12-2). Observations are detailed below:

At the immediate footwall to the mineralized zone shale units were regularly observed. Sometime these were logged as such in the primary lithology field, and sometime they were captured in the alteration/structure or comments fields of the database. The shale units are generally very sheared. Where the shale units correlate with mineralization it is possible the low hardness of the shale compared to the sandstone and tuff created a zone of weakness where shearing and subsequent mineralization has occurred. They are generally devoid of veining, so any mineralization is associated with the shale itself, but whether it pre or post-dates the shearing is not known.

From a competency point of view, there appear to be two reasons the footwall is less competent then the tuff. The tuff has generally the same composition/provenance as the sandstone footwall (felsic volcanic), but it is obviously more brecciated in texture. This has allowed the silica and chlorite alteration to penetrate the tuff more. This in turn has increased its hardness. In comparison, the footwall is composed of layers of sandstone, siltstone and shales. These individual rock types are more compacted than the tuff and therefore have been less altered, except between the rock types where the differences in grain size is more pronounced. This has allowed regular shearing along the lithology boundaries within the footwall to occur (particularly where the shale is located) and therefore a less component footwall. This variation between sandstone, siltstone and shale doesn' t appear to have been logged consistently, but it would be of significant use from a mining perspective, particularly within the first 10-15m of the footwall contact.

The isolated mineralization observed outside the main corridor of mineralization was also investigated. While there were only a few instances of this in the available core, generally any mineralization was associated with isolated veining or a shear/fault zone.

Overall the logging contained in the database correlated well with the main lithological contact boundaries observed in the core. There were slight discrepancies in some areas (generally a lack of detail), but none which SRK deemed to be inappropriate for use in a resource model.

Table 12-2: Holes and intervals checked

BHID	From	To	Meters Logged
MC002	284.5	316.47	31.97
MC006	563.4	587.29	23.89
MC004	430.31	463.08	32.77
MD009	96.5	113.8	17.30

BHID	From	To	Meters Logged
MD035	435.72	477.55	41.83
MD038	408	450.3	42.30
MD042	337.25	370.8	33.55
MD027	180.1	218.77	38.67
MC005	578.33	621.41	43.08
MD057	349.9	382.5	32.60
MD051	435.65	473.65	38.00
MD045W1	490.5	520	29.50
MD039	360.8	385.03	24.23
Total			**429.69**

The logging correlated well with what was perceived to be the main lithological contact boundaries. There were slight discrepancies in some areas (generally a lack of detail).

At the immediate footwall to the mineralized zone regular shale units were observed. Occasionally they were logged as such, and sometimes it was captured in the alteration/structure. Generally the shales were quite sheared so this is understandable. Where they correlate with mineralization it is possible the low hardness of the shale compared to the sandstone and tuff created a zone of weakness where shearing and subsequent mineralization has occurred. They are generally devoid of veining, so any mineralization is associated with the shale itself, but whether it pre or post-dates the shearing is not clear.

From a competency perspective, there appear to be two reasons the footwall is less competent then the tuff. The tuff has similar composition/provenance as the sandstone footwall (felsic volcanic), but it is obviously more brecciated in texture. This has allowed the silica and chlorite alteration to penetrate the tuff more pervasively. This has subsequently increased its hardness. In comparison, the footwall is composed of layers of sandstone, siltstone and shales. These individual rock types are more compacted than the tuff and therefore have been less altered, except between the lithologies where the difference in grain size is more pronounced. This has allowed regular shearing along the lithology boundaries within the footwall to occur (particularly where the shale is located) and therefore a less component footwall. This variation between sandstone, siltstone and shale doesn' t appear to have been logged consistently, but it would be of significant use from a mining perspective, particularly within the first 10 - 15 m of the footwall contact.

The isolated mineralization observed outside the wireframed 0.1 g/t halo was also investigated. Only a few instances of this were available at the core shed to inspect, however, the few examples seen suggest that generally any mineralization is associated with isolated veining or a shear/fault zone. T

he information gathered from the validation logging was then cross checked thoroughly with the database, and SRK believe the geology model created was appropriate for use as the basis of the resource estimate. Any variations observed with the logging will be used to better understand the genesis of the deposit and help design a future infill drill program.

12.2 2011 Newmarket Gold soil sampling program

Newmarket Gold utilize specialized industry computer software to manage its drillhole and assay database and employ dedicated personnel to manage the database and apply appropriate QAQC procedures to maintain the integrity of the data. Data is assessed for errors against standards and blanks prior to loading into Maxwell GeoServices Datashed™ database software. Data is then spatially assessed in commercially available mining software package Micromine™ for any other questionable results.

Previously, consultants have completed various database checks, which have not identified any reportable errors, which would have raised any concerns about the integrity of the data. During the preparation of this report, which has included search and lookup of assay results, generation of plans and sections and estimation of Mineral Resources, the Qualified Persons did not encounter any difficulties with the database; SRK believes the historical data/database has been verified to a sufficient level to permit its use and confidence in its reliability.

Wherever possible Newmarket Gold has also conducted on ground checks of data, this includes the- re surveying of historic drill collars and previously mined open pits. The checking of the open pits has involved the use of a surveyor with a depth sounder to test the bottom of the pit against previous pit pickups. This was done to ensure an accurate depletion of the Mineral Resource.

During the past 2 years Newmarket Gold has spent a large amount of time and money reviewing all historic data in both hard and soft copy forms. This has given the Company a much better understanding of the original data that is available for cross checking and review.

12.3 Sampling prior to 2011

Access software was implemented to manage the Maud Creek database in 2006. The software includes a strict, controlled and structured set of fields and columns to manage the data flow, and checks to alert the database manager of any data importation issues.

The geological interpretation, core logging facility and core storage areas were inspected by Snowden in 2006. In all instances the lithologies, mineralization, alteration and sample intervals were found to agree with the drill logs.

Snowden reviewed the database and confirmed that the data extracted for resource estimation matched the primary database records. Overall the review in 2006 concluded that the data has been verified to a sufficient level to permit its use in a CIM compliant resource estimate.

In 2006 Snowden reviewed all previous drilling data and concluded that the lack of documented and relevant QAQC data and protocols was material to the previous estimate, and until addressed would impact upon the ability to classify the resource estimate with greater confidence than Inferred. Based on this advice a resampling program was implemented by GBS, whereby remaining core was resampled and assayed together with the submission of independent certified reference materials (CRMs).

In July 2005 179 previously cut and analyzed core intervals were resampled and submitted to SGS laboratory in Perth for analysis along with a number of CRMs. The results of the program are detailed in the Technical Report Maud Creek Project Drill-Hole Data Validation for Resource Assessment by Andrew Milne of GeoCraft Pty Ltd dated August 2005. Standard fire assay analysis was undertaken on the 179 samples, and then 57 samples were re-assayed using screen fire techniques to compare against the corresponding fire assay analysis.

The following concerns regarding the program were identified:

- Sampling and analysis of different parts of the core,
- Different proportions (half vs a quarter) of the core in some cases; and
- Different Laboratories performing the initial (ALS or Assay Corp) and Resample (SGS) analysis.

Figure 12-1 and Figure 12-2 are precision plots comparing the original data vs the resample and the fire assay data vs the screen fir e data, respectively. Figure 12-1 shows about 30% of the data plotting above the 20% precision line, which given the concerns raised above is an acceptable level.

Figure 12-2 suggests that coarse gold is not affecting the fire assay results and that they can be considered acceptable for use.

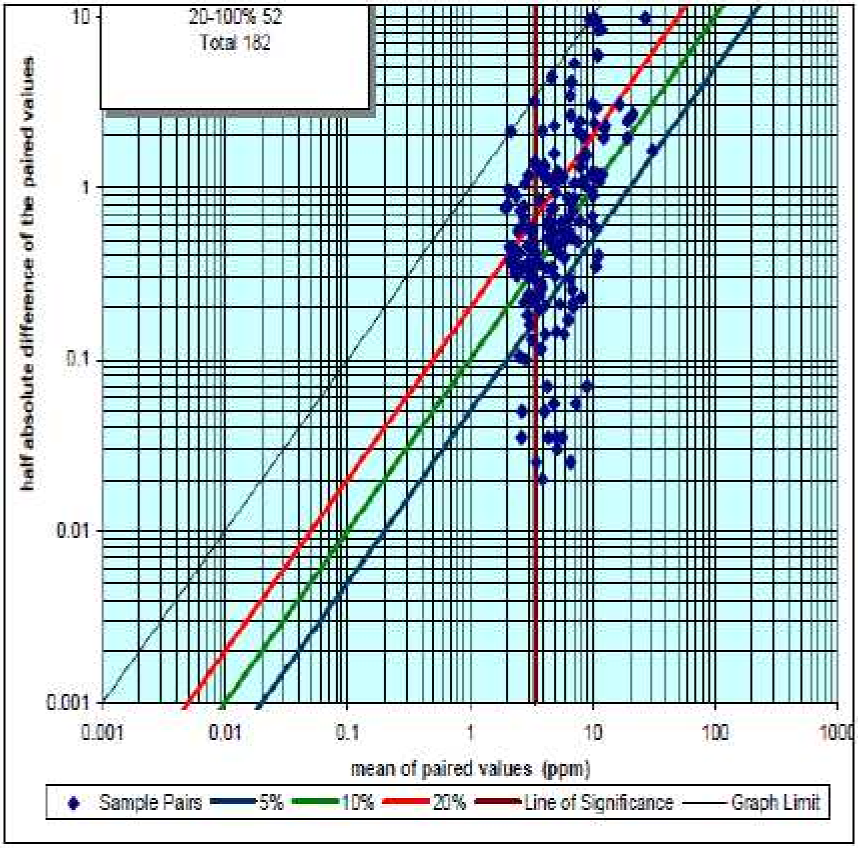


Figure 12-1: Precision plot original vs resample

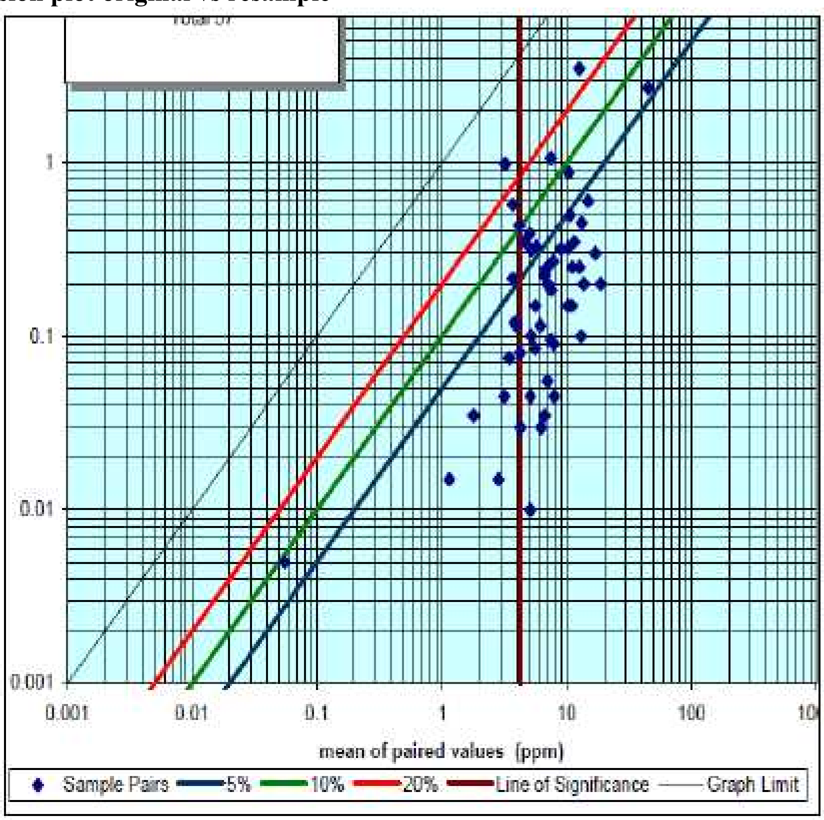


Figure 12-2: Precision plot fire assay vs screen fire assay

13 Mineral Processing and Metallurgical Testing

13.1 Metallurgical Testing

An extensive program of metallurgical testing was carried out from 1994 through to 2006. Much of the focus and testing was on downstream oxidation processes on refractory mineralization, such as BIOX® and the GEOCOAT® Process. This summary of metallurgical testing considers only those parts relevant to the current flowsheet selection, i.e.:

- Crushing and grinding
- Gravity recovery
- Flotation
- Tailings and concentrate dewatering

Direct cyanidation leaching of mineralization and concentrates was tested on the fresh (sulphide) mineralization with poor results and is omitted from this summary.

The major body of work was undertaken for Kalmet Resources N.L. by Ammtec in 1996 – 1998. Nine separate reports focussed on flotation testing, while only one dealt directly with SAG milling.

SAG milling was not tested in detail because the previous owners envisaged a relatively low throughput processing facility (300 ktpa) and using a conventional crushing plant that would suitable for both an oxide mineralization gold heap leach circuit as well as a crushed product size suitable as mill feed. The lack of SAG mill testing is not considered an issue because processing is through the Union Reefs which incorporates three stage of crushing and closed circuit ball milling.

13.1.1 Comminution

Three reports cover measurements of physical parameters for crushing and grinding:

- Ammtec report A5161 "Metallurgical testwork on variability samples VL 5-8 and VF 1- 8 from the Maud Creek Project for Kalmet Resources" (December 1996)
- Ammtec report A6076 "Metallurgical testing of variability samples VF9 – VF15 from the Maud Creek Gold Project for Kilkenny Gold NL" (February 1998)
- Ammtec report A6443, "SAG milling testwork associated with the Maud Creek Gold Project for Kilkenny Gold NL" (October 1998).

Crushing

Reports A5161 and A6443 include measurements of crushing work index (CWi) and unconfined compressive strength (UCS).

Report A5161 describes samples VL 5 – 8, selected from hole MD20 intervals to represent a profile of increasing depth and sulphur level (less oxidised). Ten specimens were selected from these intervals for crushing work index tests and 5 specimens were selected for UCS tests. Results are as presented in Table 13-1.

Table 13-1: Crushing Test results from Report A5161

Sample	Drillhole	Depth (m)	Ore type	CWi (kWh/t)	Lithology
CWi-1	MD21	28.5	Oxide/transition	16.2	Oxidised tuff, quarts & carbonate
CWi-2	MD21	32.7	Transition	8.5	Breccia tuff, quartz carbonate, graphite
CWi-3	MD21	41.5	Transition	11.1	Massive quartz, graphitic, minor pyrite & arsenopyrite
CWi-4	MD20	67.3	Primary	8.2	Graphitic quartz carbonate stockwork, arsenopyrite
CWi-5	MD20	82.0	Primary	10.9	Massive quartz, pyrite, arsenopyrite,
CWi-6	MD14	86.1	Primary	13.5	Tuff, weak stockwork, pyrite
CWi-7	MD14	92.0	Primary	10.0	Tuff
CWi-8	MD14	105.1	Primary	8.5	Massive quartz vein, minor graphite
CWi-9	MD14	109.2	Primary	8.9	Quartz breccia, graphite, pyrite, arsenopyrite
CWi-10	MD14	112.5	Primary	5.9	Graphite quartz stockwork, pyrite, arsenopyrite
			Average	**10.2**	
			Maximum	**16.2**	
			Minimum	**5.9**	
			75th percentile	**11.05**	

The UCS results ranged from 52 – 285 MPa with an average of 163 MPa.

Report A6443 describes the samples tested as from '5 trays of recently drilled HQ core' . Intervals are specified but the exact drillhole was not identified. The core includes oxide, main lode and hanging wall intervals. Four sulphide core specimens were tested for UCS; three from the main lode and one from the hanging wall. Sixteen sulphide core specimens were tested for crushing work index (CWi). Results are as follows, they exclude the oxide sample test results shown in Table 13-2.

Table 13-2: Crushing Test Results from Report A6443 (Oxides Excluded)

	CWi (kWh/t)	UCS (MPa)
Average	10.7	132
Maximum	23.0	182
Minimum	4.5	106
75th percentile	13.2	139

These results indicate moderate average power requirements, but with wide variation. The UCS results would be classified as strong (60 – 200 MPa), with the maximum result approaching the very strong level (>200 MPa) but is not at levels that cause crushing difficulties with appropriate equipment selection. An additional sample VL5-8 had additional UCS tests undertaken on it. The deepest sample demonstrated very high competency (285 MPa) and a 75th percentile of 208 MPa. The combined UCS 75th percentile used for the process design criteria (PDC) is 182 MPa.

The average CWi is only moderate in strength but the maximum level is would be classified as strong. In both cases (UCS and CWi), this suggests more tests would be required to get a reliable average. In the absence of further data, a conservative value has been chosen for design. The crushing testwork results confirm that the Union Reefs three stage crushing circuit is capable of processing the hard Maud Creek fresh mineralization. It may be beneficial to blend the softer Maud Creek oxides with the Cosmo underground ores if there are any material handling or viscosity concerns.

Grinding

Ammtec report A5161 describes composite samples VF4/6 being subjected to measurements of abrasion index and bond rod and ball mill work indices (BRMWi and BBMWi). The composite was a 50/50 blend of primary mineralization sample VF4 and primary footwall sample VF6:

VF 4: Drillhole MD 20, (61.0 - 74.0 m, 80.0 - 83.7 m)

VF 6: Drillhole MD 14, (98.9 - 116.5 m).

These samples had quite high gold grades of 10.83 g/t and 12.31 g/t respectively.

Ammtec report A6076 describes the BBMWi testing of composites VF10 and VF14. The sample origin is provided as:

VF 10: Drillholes MD 40 (323.3 – 329.8 m), MD 41 (314.55 – 319.0 m), MD 44 (325.0 – 337.0 m)

VF 14: Drillholes MD 35 (459.4 – 464.8 m), MD 37 (463.0 – 469.1 m), MD 38 (429.0 – 436.0 m)

Results from the two reports are summarised in Table 13-3.

Table 13-3: Grinding Test Results from Reports A5161 and A6076

	Average depth (m)	BRMWi (kWh/t)	BBMWi (kWh/t)
Earlier work		16.65	17.73
VF4/6	78	17.8	18.1
VF10	283		18.6
VF14	392		19.64
Average		19.0	18.8

Specific grinding power consumption was also recorded in the flotation pilot plant runs:

1996 pilot run: 12.24 kWh/t

1997 pilot run: 11.95 kWh/t

The equipment and feed size were not in accordance with the standard Bond work index methods, so the results should be treated with caution, however they are indicative. In summary, all the comminution test programs indicate a moderately hard, highly abrasive mineralization requiring high grinding energy and highly variable crushing energy.

Summary data from John MacIntyre' s comminution evaluation report are presented in Table 13-4.

Table 13-4: Summary of Comminution Results from J MacIntyre Metallurgical Evaluation (Sept '98)

		Oxide	Primary
Bond crushing work index	kWh/t	5.0	10.2
Bond Rod mill work index	kWh/t	16.65	19.0
Bond Ball mill work index	kWh/t	17.73	18.8
Abrasion index		0.489	0.678
Unconfined compressive strength	MPa	49	163

The following observations are reproduced from the MacIntyre report:

1. *The average Bond crushing work index values for both the oxide and primary zones are low. Although the database is limited the primary zone crushing work index does not appear to increase with depth to a vertical depth of 100 m.*

2. *Both the oxide and primary zone rod and ball mill work index values are high. No rod mill work index value has been measured on samples obtained from a depth greater than 78 m. The primary zone ball mill work index values do not appear to be depth sensitive to depth of 392 m.*

3. *The oxide abrasion index is above average and the primary zone abrasion index is high.*

4. *The unconfined compressive strength (UCS) values are strongly dependent on depth, increasing from 37 MPa at 5 m depth to a very high value of 285 MPa at 90 m depth. No UCS values have been measured on samples obtained from a depth greater than 90 m.*

Subsequent to the MacIntyre report, Ammtec report A6443 described abrasion index (Ai), milling work index and JK drop weight tests on a "comminution composite sample" made up of HQ core from the main lode and hanging wall (oxides excluded). As with the crushing work, the intervals are specified and come from a single, unidentified drillhole. Results were very similar to the earlier reports as shown in Table 13-5.

Table 13-5: Comminution Results from Report A6443

	Ai	BRMWi (kWh/t)	BBMWi (kWh/t)
Maud Creek 'Comminution composite'	0.6487	18.3	19.94

The same composite sample was used for JK Drop Weight tests, giving the following results:

A: 72.8
b: 0.68
Axb: 49.5
ta: 0.41

These parameters define the ore-specific breakage function which can be input to the JK simulation software to predict SAG mill performance and sizings. The tests show this sample to be moderately hard.

The grinding testwork results confirm that the Union Reefs three stage crushing and closed circuit ball milling circuit is capable of processing the hard Maud Creek fresh mineralization. There is a large amount of installed milling power to meet the required grinding power demand. Future tests should be based on geo-metallurgical domains in order to match grinding requirements with the mine plan and to ensure sample representivity for the new mine plan.

Relatively conservative comminution parameters have been incorporated into the design criteria. No significant risks are considered in this aspect of the testwork. There is sufficient comminution capacity at Union Reefs. Additional testwork is not considered to be essential but would provide further confidence and optimisation of the design. Specialist comminution modelling of the Maud Creek mineralization through the Union Reefs circuit is recommended during the next phase of study to confirm the capacity, likely throughput rates and improve confidence in the operating costs.

13.1.2 Gravity Gold Recovery

Gravity Recoverable Gold (GRG) testwork was included in the three flotation pilot plant runs as well as several of the batch testwork programs.

Several preliminary programs were conducted by Amdel and Metcon in 1994. Gravity recovery was found to be significant but highly variable.

Ammtec report A4997 "Optimisation flotation testing of Maud Creek primary gold ore for Kalmet Resources" (May 1996) describes flotation and gravity testing of a composite prepared from drillhole WD 16, 122 – 144m. Batch tests were done in a Knelson (gravity) Concentrator after crushing the sample to 100% passing 1mm, giving 11.36% Au recovery.

Ammtec report A5161 "Metallurgical testwork on variability samples VL 5-8 and VF 1-8 from the Maud Creek Project for Kalmet Resources" (December 1996) included gravity pre-treatment by Knelson Concentrator on samples designated VF 1 – 8. Gravity gold recovery varied from 2.05% to 74.05% on the VF5 sample, noted as being high in carbonates. Summary results are provided in Table 13-6.

The high carbonate sample VF5 was further investigated in Ammtec report A6260 "Flotation optimisation work associated with the Maud Creek gold project for Kilkenny gold NL" (June 1998). As previously demonstrated, very high gravity recovery was observed. A 65.7% recovery of feed gold was recorded from a Knelson concentrator treating P_{80} 500 micron mineralization.

The first pilot plant operation is described in Ammtec report A4952 "Pilot scale flotation testing of Maud Creek primary gold ore for Kalmet Resources" (March 1996). The bulk composite used had a relatively high gold head grade of 9.88 g/t (average). Gravity gold recovery equivalent to 14.5% of the feed was reported. There was also free gold found in the flash flotation concentrate, but this was deemed too fine to be gravity recoverable.

The second pilot plant also included gravity recovery, as described in Ammtec Report A5367 (Part A), "Pilot scale flotation testing of Maud Creek primary ore for Kalmet Resources NL" (March 1997). Gravity recovery equivalent to 14.5% of the feed ore was reported.

Ammtec report A6076 "Metallurgical testing of variability samples VF9 – VF15 from the Maud Creek gold project for Kilkenny Gold NL" (February 1998) included a series of gravity recovery tests. Unlike previous tests, results were generally poor. However, the gravity tails from these tests all responded well to flotation and overall recoveries remained high. A summary table is provided in Table 13-7.

The J MacIntyre report (Sept 98) makes the following observations in respect to gravity gold recovery:

1. *The amount of gravity gold recovered generally increases with gold head grade. A minimum amount of gravity gold may be recovered for gold head grades generally less than 5.6 g/t.*
2. *The first and second pilot plants recovered …15.8% average….of the total gold.*
3. *The first and second pilot plants recovered 1.430 g/t and 0.975 g/t (1.203 g/t average) of their head grades of 7.837 g/t and 7.373 g/t (7.605-g/t average). That is 18.2% and 13.2% (15.8% average) of the total gold that was recovered as amalgam gold.*
4. *The amalgam gold-gold head grade relationship predicts that approximately 1.55 g/t or 20.4% of the gold is recovered as amalgam gold for the average pilot plant head grade of 7.605 g/t. This relationship has been adjusted downwards by 0.35 g/t such that it reflects the actual average amount of gold recovered by both the pilot plants. The adjusted relationship therefore predicts that 1.049 g/t or 14.1% of the total gold to be recovered as amalgam gold for a 7.40 g/t head grade.*
5. *The amount of amalgam gold recovered appears to decrease with depth, especially when samples grading more than 10 g/t are excluded from the database.*
6. *The amount of amalgam gold recovered appears to be independent of whether the sample is oxide, transition or primary mineralization.*
7. *The amount of amalgam gold recovered appears to be independent of either the sulphur or arsenic head grade.*

Some later testwork also assessed gravity recovery. IMO project No. 1930 “Scoping testwork on Maud Creek gold ore samples for Harmony gold operations Ltd” (June 2003) included gravity separation of a low (1.75 g/t) and high grade (6.14 g/t) composite. The composites were prepared from intervals of main zone (MZ) and eastern shear (ESZ) from a several different drillholes. Gravity recovery to low grade concentrates was reported as 44% from the low grade and 53% from the high grade. These results are probably not realistic due to the high mass pull reporting to the gravity concentrate but support the inclusion of a gravity recovery circuit and potential for reasonable gravity recovery on grades more likely to be fed to the Processing Plant.

The most recent pilot program is covered in Ammtec report A9911 “Pilot flotation on Maud Creek deposit for Terra Gold Mining Ltd” (February 2006). Composites samples were prepared from drillholes MD63 and MD65. Gravity recovery of 18.9% was recorded from a head grade of 1.94 g/t.

In summary, the Maud Creek contains significant but varying amounts of gravity recoverable gold. The relationship presented by John MacIntyre is as follows:

g/t amalgam gold = 0. 760 x (gold head grade) - 4.58

Based on the relationship above, the amount of gravity recoverable gold would be minimal (negative) for an average head grade of 4.38 g/t. This is not supported by much of the testwork, there are significant variations in recoveries in the various testwork that has been completed, and the value and relationship given in the John MacIntyre report is considered too conservative.

A gravity circuit would typically be included in a new plant design for a GRG content above 10% and in this case is further supported given the cost of transporting a flotation concentrate for third party processing. Union Reefs already has a gravity gold circuit and therefore the decision on whether to include it in the design is not relevant. In summary, it is recommended that further gravity recovery testwork be performed on mineralization that more closely reflects the current average head grade. It would help better define the payable gold attributed to the gravity concentrate and the flotation concentrate.

Until further assessment is made on the deposit, a gravity recovery of 20% has been used for design purposes based on variability and pilot plant results. This will be updated after further review in the next stage of study. If gravity recovery proves to be lower than this, testwork demonstrates gold is subsequently recovered in flotation, i.e. overall recovery is relatively robust against variation in the gravity gold recovery.

13.1.3 Flotation

Eleven separate testwork reports describe the flotation programs undertaken for Maud Creek. All were carried out by Ammtec Laboratories. The reports can be separated into several phases of work, being; preliminary, variability and optimisation testwork followed by pilot programs. A significant amount of flotation testwork has been undertaken and is considered to support the PEA. Piloting would normally be considered to be at a feasibility level of assessment. This testwork is the key to the Union Reefs flotation plant upgrade.

Preliminary programs

Ammtec Report A4909 “Preliminary assay and flotation testing of a Maud Creek ore composite for Kalmet Resources” (January 1996)

Ammtec Report A4930 “Preliminary metallurgical testwork on Maud Creek primary gold ore composite for Kalmet Resources N.L.” (January 1996)

The first of these (A4909) was a simple flotation test on sample from hole (MD 09) with head grade 7.16 g/t Au. Flotation recovery was very high at 95.55%, to a concentrate containing 128.0 g/t Au.

Cyanide leach extraction on the tail gave only 55.1% recovery.

The second program (A4930) was a single set of rougher flotation batch tests to confirm the suitability of the sample for the first pilot program (A4952). The head grade was quite high (11.1, 8.66 g/t Au). In this sample 94.7% of gold was recovered to a concentrate containing 53.9 g/t Au.

The grind size for both of these test programs was 80% passing 75 microns.

Optimisation programs

The next set of tests are considered optimisation batch testwork programs.

Ammtec report A4997 "Optimisation flotation testing of Maud Creek primary gold ore for Kalmet Resources" (May 1996)

Ammtec Report A5376 Part B "Maud Creek carbonate depression flotation testwork for Kalmet Resources" (February 1997)

Ammtec A6260 "Flotation optimisation work associated with the Maud Creek gold project for Kilkenny gold NL" (June 1998)

Ammtec report A9617 "Flotation testwork on Maud Creek sample TMCD4002 for Terragold Ltd" (May 2005)

The first program aimed to optimise the grind size, circuit configuration, and reagent scheme. The sample was taken from drillhole WD 16, 122 - 144 m, with an average head grade of 3.73 g/t Au. The optimum reagent scheme was reported to be:

125 g/t SIBX (collector)

40 g/t AP3477 (collector)

50 g/t CuSO4 (activator)

However the performance was not sensitive to either the type or dose of reagent over the ranges tested.

Flash flotation caused only a marginal increase in gold recovery. Cleaner flotation was not included in the program. At the time of testing it was not considered to be necessary.

Grind sensitivity size tests showed a gradual drop in gold recovery with increasing grind size to flotation. From this data, a P_{80} of 75 microns was selected for subsequent testwork. However this is based on a single drillhole and may not be repeatable.

Report A5376 describes an unsuccessful attempt to depress flotation of carbonates using proprietary reagents. Carbonates are mainly an issue if BIOX® processing is used downstream, where acid can be a major cost.

Report A6260 focussed on the effect of downstream BIOX® processing on flotation. Specifically, using acidic water for flotation and using flotation tails for neutralization of BIOX® liquor. There was also some extra flotation testing on the high carbonate sample VF5. High overall recovery was found to be possible from closed cycle rougher/scavenger tests, but with some sacrifice in concentrate grade.

Report A9617 used intervals from drillhole TMCD04002, 192 - 208 m. Batch flotation tests were done on a composite and also on three separate drillhole intervals with varying gold grades. The composite gave very good flotation performance; with a 95.3% gold recovery to a 190 g/t concentrate.

Recovery by grind size showed no improvement from 106 microns down to 75 micron. This is in contrast with the previous results of A4997. Grind size optimisation between 75 and 150 microns.

The process design criteria has selected a size of 75 microns for recovery and concentrate quality purposes and to reflect pilot plant parameters. There may be justification in relaxing this grind size target marginally in future assessments. Union Reefs capacity is sufficient for any of these grind size target options therefore it can be optimized after start-up. This grind size flexibility is an advantage.

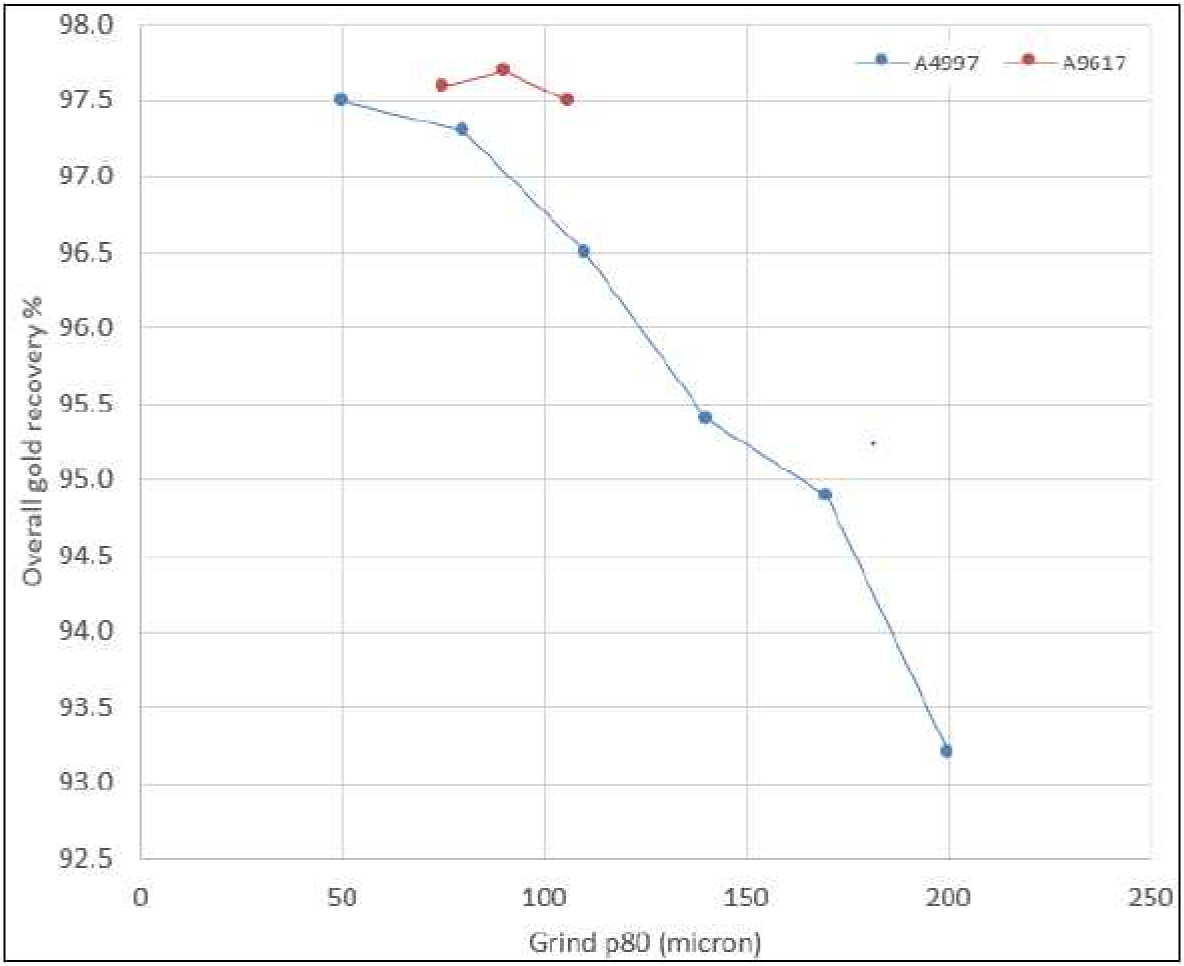


Figure 13-1: Recovery by Grind Size

Source: Extracted from Ammtec Reports A4997 & A9617

Variability programs

- Ammtec report A5161 “Metallurgical testwork on variability samples VL 5-8 and VF 1- 8 from the Maud Creek project for Kalmet Resources” (December 1996).
- Ammtec report A6076 “Metallurgical testing of variability samples VF9 - VF15 from the Maud Creek gold project for Kilkenny Gold NL” (February 1998)

The first of these tested samples taken from varying depth through the mineralization profile, testing first gravity then flotation on the gravity tails. Results are shown below in Table 13-6.

Table 13-6: Flotation Results from Ammtec Report A5161

Sample composite	Ore zone	Drillhole	Average depth (m)	Gold grade (g/t)	Gravity recovery (%)	Flotation recovery (%)	Overall recovery (%)
VF1	Oxide	MD 21	41	15.35	20.91	70.73	91.64
VF2	Oxide/ transition	MD 21	33	7.63	63.75	29.94	93.69
VF3	Trans/primary	MD 21	33	12.72	41.01	55.64	96.65
VF4	Primary	MD 20	61	10.83	17.24	79.68	96.92
VF5	Primary hanging wall	MD 14	76	7.76	74.05	24.56	98.61
VF6	Primary foot wall	MD 14	93	12.31	46.07	52.06	98.13
VF7	Primary	MD 3	121	16.09	70.73	28.43	99.16
VF8	Primary	MD 27	175	3.36	2.05	93.23	95.28
Average					**41.98**	**54.28**	**96.26**
Maximum					**74.05**	**93.23**	**99.16**
Minimum					**2.05**	**24.56**	**91.64**

Overall recoveries were consistently high, although the gravity /flotation proportion varied widely. The second variability program, A6076, tested samples VF9-VF15. Flotation recovery on ore was consistently over 95%, with one exception, VF12 at 85.65% . The gravity recovery was significantly lower with only one sample demonstrating any notable GRG. These samples were significantly deeper than the previous optimisation tests.

Table 13-7: Flotation Results from Ammtec Report A6076

Sample composite	Drillhole ID	Average depth (m)	Gold feed grade (g/t)	Gravity recovery (Au%)	Float recovery (Au%)*	Overall recovery (Au%)
VF9	MD 038	298.7	5.94	0.47	95.44	95.46
VF10	MD 040 MD 041 MD 044	326.6 316.8 331.0	5.96	11.99	95.47	96.01
VF11	MD 036 MD 045	290.6 438.5	14.97	0.24	94.67	94.68
VF12	MD 035	383.7	5.89	0.53	85.65	85.73
VF13	MD 034	438.0	4.77	0.38	97.15	97.16
VF14	MD 035 MD 037 MD 038	462.1 466.1 432.5	3.90	1.94	94.85	94.95
VF15	MD 036 MD 045	393.6 474.8	5.14	1.87	96.67	96.73
Average				**2.49**	**94.27**	**94.39**
Maximum				**11.99**	**97.15**	**97.16**
Minimum				**0.24**	**85.65**	**85.73**

*Flotation recovery% from gravity tail

There appears to be a grade versus recovery relationship as shown below in Figure 13-2; however, the correlation co-efficient is poor. Even removing the main outlier does not significantly improve it. At the expected feed grade of approximately 4.38 g/t as per the design criteria, the overall recovery of 95% is considered to be conservative. Further optimisation of the grade recovery relationship (and if possible integration with product grade) under the optimised conditions should be undertaken at the next stage of study.

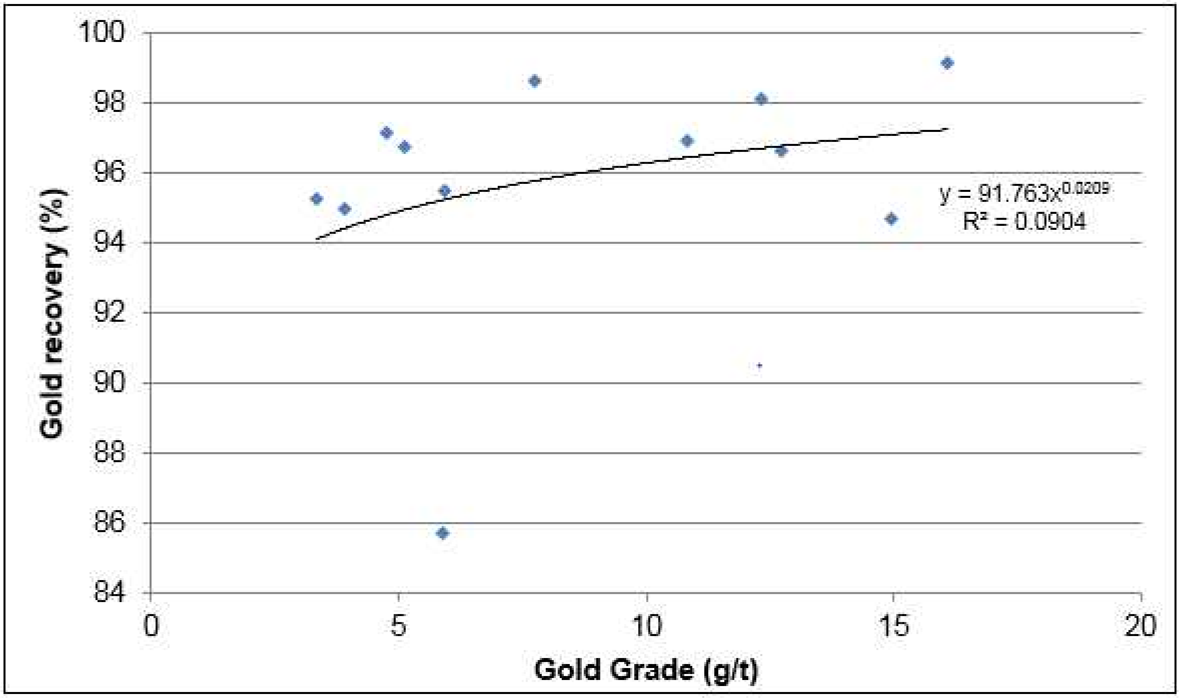


Figure 13-2: Recovery vs. Grade

Source: Ammtec tests A5161 and A6067

Pilot plant programs

Three flotation/gravity pilot plants have been run. In each case, the main purpose was to generate concentrate for BIOX® testing (normally it would be difficult to justify this level of testing for just a conventional concentrator only). The results also provide good process design data.

Ammtec report A4952 "Pilot scale flotation testing of Maud Creek primary gold ore for Kalmet Resources" (March 1996)

Ammtec Report A5367 (Part A) "Pilot scale flotation testing of Maud Creek primary ore for Kalmet Resources NL" (March 1997)

Ammtec A9911 "Pilot flotation on Maud Creek deposit for Terra Gold Mining Ltd" (February 2006)

The first pilot program used a 5 tonne composite sample. The head grade was quite high, with assay measurements of 11.1 and 8.66 g/t Au. Sulphur grades were also quite high at 2.59% .

The pilot testwork circuit included closed circuit grinding with flash flotation and mill discharge passed over a corduroy cloth to collect 'gravity' gold. Cyclone overflow passed to rougher, middling and scavenger flotation cells. The reagent scheme consisted of:

50 g/t of copper sulphate added to the mill;

150 g/t of collector SEX stage added as the collector; and

10 g/t of frother MIBC stage added.

The flotation feed P_{80} was 118 microns, while most concentrate was reground to 17 microns. Gravity recovery was reported as 18.2%, 92.1% from flotation of the gravity tail and 93.5% overall. The flotation concentrate grade contained 51.4 g/t gold and 19.5% sulphur. The overall concentrate gold grade was 63.82 g/t gold.

Flotation gold recovery was 3.5% lower than expected from bench scale tests. This was attributed to the very fine nature of the RC drill chips used.

Flash flotation recovered 41% of the gold, 44% of the sulphur and 38% of the arsenic into 4.0% of the mass.

The second pilot run in 1997 used a bulk sample of lower grade material taken from RC chips, with a lower average Au grade of 6.69 g/t and 1.85% sulphur. Flotation feed P_{80} ranged from 87 to 100 microns. Flotation recovery varied from 87.3% to 94.8%, with higher recovery corresponding to lower concentrate grade. This supports the future development of the feed and product grade versus recovery relationship once a more extensive data set is available. Combined results from the four survey points are shown in Figure 13-3.

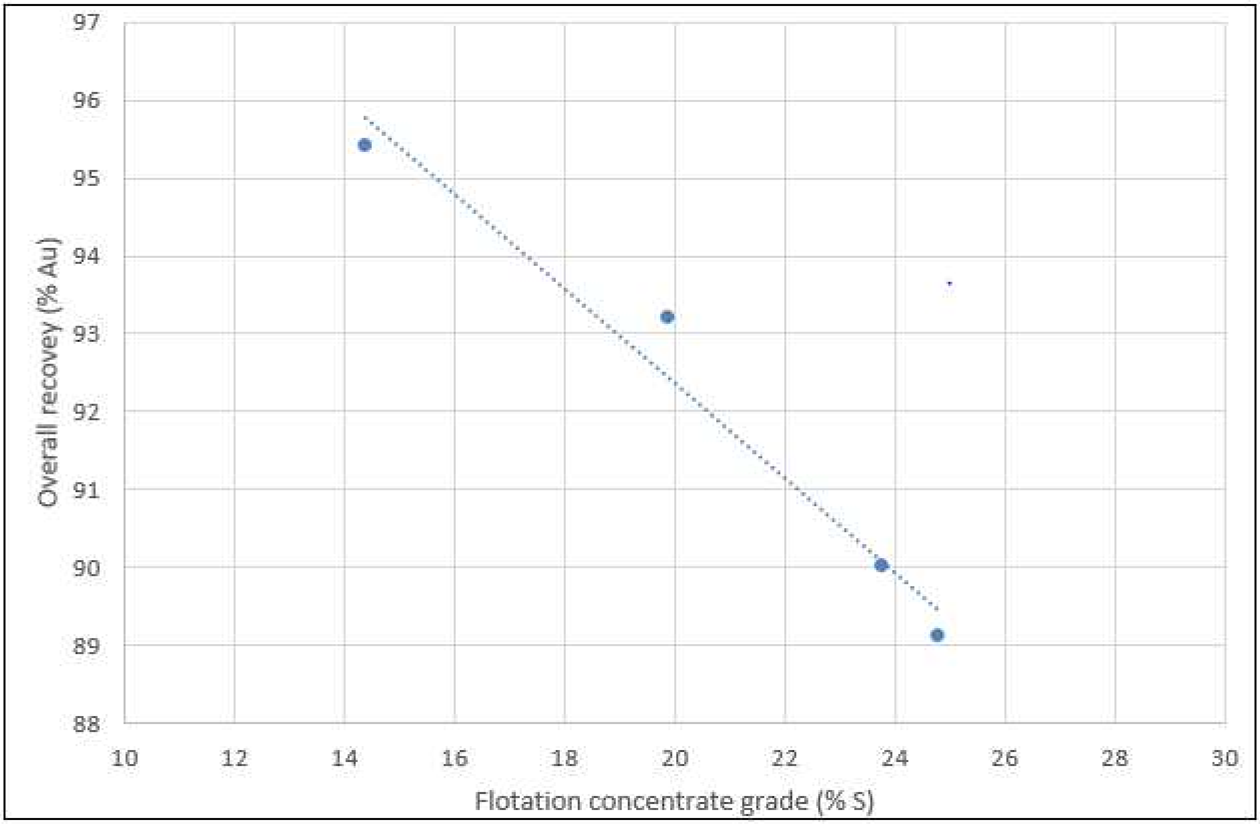


Figure 13-3: 1997 Pilot Plant Survey Results

The lower two recovery points included cleaner flotation while the higher recovery points did not. The cleaner stage was added to increase sulphur grade to over 18 - 20%. It also raised the gold grade from 47.3 g/t to 69.4 g/t. The following points regarding the second pilot plant are extracted from the J MacIntyre evaluation report:

The pilot plant consisted of a gravity concentration stage, a flash flotation stage and a secondary flotation stage. Total flotation residence time was 46 minutes;

Mill feed was crushed to a P_{100} of 4.0 mm and also had a very fine P_{80} crushed product size of 1.2 mm;

A bulk secondary float was employed for the first two days of the pilot plant. A cleaning stage was used on the middling and scavenger concentrates for the last two days; and

The plus 180-micron fraction of the flash flotation concentrate (approximately 8.7- kg wet) was the only product reground to a target P_{100} of 180 microns. The following points are concluded.

1 *An excellent reconciliation exists between the calculated head grade and the drillhole head grade. The calculated head grade of 7.37 g/t agrees within 1% of the drillhole grade of 7.41 g/t.*

2 *A total gravity plus flotation recovery of 91.9% was realised. The first two days recovery of 94.3% was achieved without any cleaning stages – this produced a saleable concentrate grade. This reduced to 89.6% for the last two days when a cleaning stage was employed on the middling and scavenger concentrates.*

3 *13.2% of the total gold was amalgamated gravity concentrate gold.*

4 *93.0% of the gravity tail gold was recovered by flotation in the first two days. This is 3.2% less than the 96.2% gravity tail recovery predicted from Section 2.9.3's grind-recovery relationship. The very fine nature of the RC drill chip would have also contributed to the much lower than predicted recovery.*

5 *Flash flotation recovered 45% of the gold (41% for the first pilot plant), 50% (44%) of the sulphur and 41% (38%) of the arsenic into 3.8% (4.0%) of the mass.*

6 *The sulphur recovery for the first two days of 99.0% is similar to the predicted sulphur recovery of 98.5%. Sulphur recovery reduces to 95.6% when a cleaning stage was employed for the last two days.*

7 *The arsenic recovery of 90.1% for the first two days is similar the 89.4% predicted from Section 2.9.3's grind-recovery relationships. Arsenic recovery reduces to 85.7% when a cleaning stage was employed for the last two days.*

8 *The second pilot plant sample contains 205 ppm of copper, of which 84% is recovered into the combined flotation concentrate at a mean grade 1,801 ppm.*

9 *Days 3 and 4's cleaned concentrate contains 4.4% of the carbonate being 54% of the amount of carbonate contained in Day 1 and 2's un- cleaned concentrate of 8.2%.The cleaned concentrate mass of 7.0% is also 57% of the value of the uncleaned concentrate mass of 12.3% . That is both the uncleaned and cleaned concentrate grades are similar at 6.0% and 5.4% respectively.*

10 *The amount of calcium contained in the concentrate is similar to the carbonate content. That is 8.0% calcium versus 8.2% carbonate for the un-cleaned concentrate and 3.6% calcium versus 4.4% carbonate for the cleaned concentrate.*

11 *The second pilot plant sample contains a negligible amount of mercury. The highest value recorded in the concentrate was 0.038 ppm.*

12 *The combined flotation concentrate's mean P_{80} of 66 microns is much coarser than the 17 microns for the first pilot plant, but is consistent with the BIOX® requirements.*

The comments regarding carbonates and BIOX® requirements can be ignored if concentrate is to be exported as per the base case. The most recent test program was conducted in 2006 as described by Ammtec report A9911 "Pilot flotation on Maud Creek deposit for Terra Gold Mining Ltd" (February 2006). Tests were conducted on historical drill core from holes MD063 and MD065, drilled in the late 1990s. The condition of the core may have deteriorated to some extent (oxidised) post drilling. The samples had not been refrigerated.

A series of bench scale tests showed high flotation rougher recovery (~93 - 94%) and effective cleaner flotation. The pilot flotation run, however, showed lower recovery:

Gravity gold: 18.9%

Flotation concentrate: 64.6%

Overall recovery: 85.5%

The grade of the final concentrate was reported as 22.2% sulphur and 40.7 g/t gold. It is not clear why the pilot run gave much poorer results than the batch tests, but the results may have been compromised by the age of the sample or the pilot operation/ stability.

13.1.4 Tailings and concentrate dewatering

Thickening tests were carried out by Supaflo (now Outotec) and BPR in support of the BIOX® process design. The more conservative results from the range below are adopted for the design:

Flotation tailings

Flocculant dose: 25 - 35g/t (M-358)

Capacity: 0.7 – 0.8 t/m^2/h

Underflow density: 68 – 70%

Flotation concentrate

Flocculant dose: 25- 35g/t (M-E10)

Capacity: 0.6 – 0.7 t/m^2/h

Underflow density: 56 – 58%

No data is available on concentrate filtration.

13.2 Future Testwork

The Maud Creek mineralization has already been subject to extensive metallurgical testing, adequate to support this PEA. However there remain some gaps that should be covered to reduce the design risk for future stages if it was decided to undertake additional testwork.

Many tests do not specify the lithological domain of the samples, and some do not specify the sample origin at all. In some cases it could be better to design around the uncertainty rather than undertake further testing. For example, the crusher could be sized for the worst-case hardness rather than attempting to determine an accurate average value.

Much of the comminution testwork was undertaken at shallower depths. It may be worth considering additional comminution testwork on deeper samples if new sample becomes available however the Union Reefs crushing and grinding circuits are adequate to meet the target throughput of the Maud Creek mineralization. At this level of study materials handling testwork is not considered to be essential.

The optimum grind size was determined for conventional crushing followed by ball milling. There may be a considerable saving in the operating cost by choosing a coarser size. This should ideally be conducted on ore-domain composites and supported by economic trade-off analysis. Some specialist comminution modelling using the Union Reefs circuit is recommended for the next stage of design.

The flotation and gravity circuits can be designed from the available results. However, confirmation of gold recovery by mineralization domain would be useful for production forecasting and economic analysis. Furthermore, some flexibility should be built in to the layout to allow for circuit changes, such as retro-fitting flash flotation and/or cleaner flotation. Operating experience and changes in the market conditions/payment terms may justify these measures to increase product grade. Further development of a feed and product grade versus recovery relationship will improve confidence in predicting the overall recovery. This should be undertaken at the next level of study.

A flash flotation circuit has not been incorporated into the design at this level of study. Limited flash testing was done over the life of the project however when tested, such as in the final piloting program, it has been shown to be relatively effective. The flowsheet has been shown to be simple and robust with good product grades without it and to keep the flowsheet simple and reduce costs; it has been excluded at this point. This aspect of the flowsheet will be considered in more detail in the next level of study as it remains a potential opportunity.

Filtration of concentrate samples should be included in future programs. These can be easily added to the flotation confirmation tests and will be valuable for sizing and selection of the filters. It is not considered a risk for design. Conservative assumptions have been made at this time.

Potential customers will most likely want to test samples of concentrate before agreeing to sales terms. Future tests should be used to generate concentrates for marketing purposes.

This study focuses mainly on metallurgy and processing of the sulphide mineralization. For the oxide and transition material (limited tonnage of transitional included in the LoM), controlled potential sulphidation (CPS) should be considered to enable flotation. CPS is an established technology currently used on gold and combined gold/ copper ores at several mines in Australia and overseas. After comminution, mineralization is treated with sodium hydrosulphide (NaHS) which reacts with the oxide minerals, rendering the surface hydrophobic and thus amenable to flotation. This could considerably increase the tonnage treatable through the flotation plant. Reagent consumption rates and recoveries would need to be established by testing.

It is noted that gravity and flotation recovery testwork on transitional mineralization is limited and the metallurgical behaviour is not particularly well understood. It only makes up approximately 230 kt of the overall LoM feed and therefore extensive testing cannot be justified, including sulphidation, but it will make up a large part of the first year' s tonnage so having a reasonable understanding of its performance, particularly the recovery is important for the Project' s cash flow. This needs further consideration at the next level of study. Discounted transitional mineralization recovery of 85% has been used for modelling purposes but there is likely to be a high degree of variability in recovery depending on the level of oxidation. There may be potential to take a flexible approach to processing the transitional mineralization through either the cyanide leach circuits at Union Reefs or even the processing through flotation with the tail processed through the cyanidation circuit if recoveries are poor.

The original focus of the Maud Creek study was on the flotation recovery of the fresh sulphide mineralization. More recently, the opportunity of processing the oxide and some of the transitional mineralization through the Union Reefs cyanidation circuit has presented itself. Additional work is required to confirm the oxide and oxide/transitional recovery through gravity and cyanidation. It only makes up a relatively small proportion of the overall feed but requires more attention at the next phase of study.

13.2.1 Delineation of Mineralization Oxidation Extent

The deposit can be classified three ways; fresh, transitional and oxide mineralization. Fresh mineralization is the primary focus of the study as it makes up the bulk of the LoM tonnage. Small amounts of transitional mineralization will be processed with the fresh mineralization, however most will report to the oxide blend. The total oxide tonnage is relatively low as the bulk of it has been previously mined. It does not justify a standalone CIL/ CIP gold plant but with the Maud Creek base case now being to process through the Union Reefs Processing Plant, the oxide mineralization can now be processed through the existing cyanide leaching facility, providing early cash flow for the Project.

Based on the classifications in Table 13-8, Zones 3 to 5 are to be treated by flotation and will drive the open pit and underground mines. Zones 1 and 2 are expected to be processed through the cyanide leach circuit but any oxide/ transitional mineralization showing poor leach recovery may be processed through the flotation circuit or through flotation and the flotation tail through cyanidation.

Table 13-8: Classification of Zones in Maud Creek Deposit assumptions

Weathering Zone	Description	Sulphur Total (%)	Metallurgical recovery (%)
1	Oxide	0.1	>90%
2	Oxide/transition	0.43	>90%
3	Transition	0.6	85%
4	Transition/fresh	1.26	95%
5	Fresh	1.24	95%

Figure 13-4 below shows the deposit and is divided into the five mineralization types, with light blue indicating Zone 1 oxide mineralization, light green indicating Zone 2 oxide/ transitional mineralization, green indicating Zone 3 transitional mineralization, yellow indicating Zone 4 transitional/fresh mineralization, and red indicating Zone 5 fresh mineralization which makes up the bulk of the overall Project tonnes. Figure 13-4 shows approximately 50m of Zones 1 to 4 mineralization covering the Zone 5 fresh mineralization.

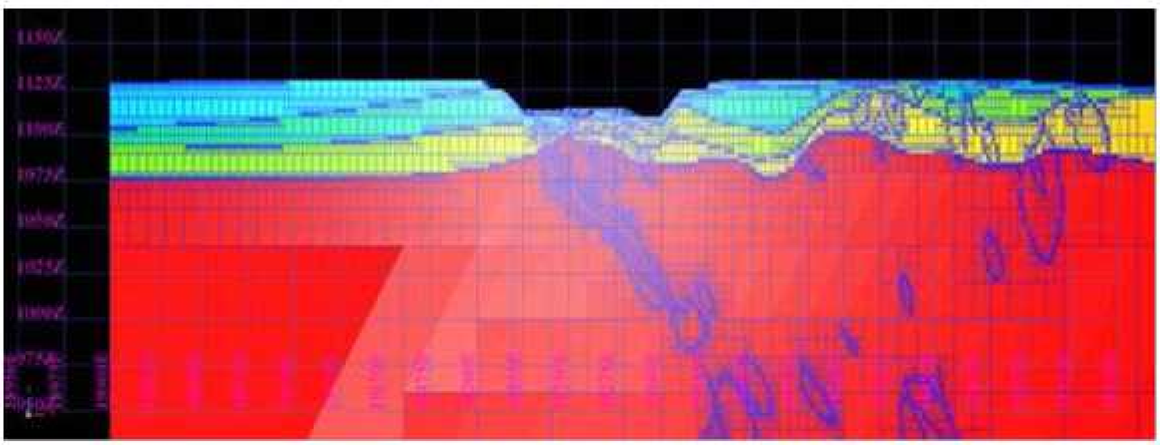


Figure 13-4: Cross section of Maud Creek Deposit

The oxides and oxides/ transitional mineralization in Zones 1 and 2 will be stockpiled separately and processed through the existing Union Reefs oxide circuit. Zones 3 and 4 are processed but have a lower transitional mineralization metallurgical recovery assigned to them.

13.3 Geometallurgy

In order to assess the impact of the Maud Creek mineralogy on reagent consumption and other geometallurgical considerations during processing, a collation of all existing metallurgical testwork was conducted (Table 13-9). Any reports which detailed the drillhole ID and from/to depth of the samples tested were incorporated into a copy of the Maud Creek drillhole assay table. To ensure there was no confusion with the existing multi-element data, any additional elemental data from metallurgical testing was given the prefix 'Met_' . All metallurgically tested intervals, including elemental and processing testwork, were assigned an identifying 'Met Sample ID' ; a combination of the year of the testwork and the 'To' depth of the sample. Where composited samples had been collected, the same value was applied for all intervals, as designated by previous assay sampling.

A total of 7,696 values were recorded from the metallurgical reports to create a geometallurgical table. These were then added to the existing 107,677 multi-element values in the assay table. The variables in the geometallurgical database identified as of interest to processing by Simulus include silver, arsenic, bismuth, carbon, carbonate carbon, CO_3, organic carbon, total carbon, sulphide sulphur, sulphate sulphur, total sulphur and antimony. Along with collar and survey information this geometallurgical data was then imported into Leapfrog and interpolations creating using the existing structural trends created for the geology model. These interpolations were then imported into Datamine and in conjunction with the drillhole database and lithology logging, amended in cross section to ensure geological considerations were taken into account.

At this point, due to limited data the carbon (carbonate carbon CO_3^{-2}, organic carbon and total carbon) were combined into one carbon wireframe interpretation, and the sulphur sulphide sulphur, sulphate sulphur and total sulphur combined into one sulphur wireframe interpretation. This exercise was conducted to determine whether there was enough data to create geostatistically robust preliminary geometallurgical domains. Unfortunately, the lack of data, presence of composite values (same value across a large interval) and preference for metallurgical testing to the south and west of the main deposit meant this was not possible.

Arsenic and silver were the only two variables with sufficient samples spread across the deposit, due to multi-element testing during previous drill programs. Arsenic was estimated in the block model using its own variography and kriging parameters, but limited to the mineralized domains previously created based on the gold samples. Due to the low numbers of silver, bismuth, carbon, sulphur and antimony data these variables were was estimated into the same gold bearing domains, rather than into their own domains. Also, due to insufficient data to generate variograms, parameters from the gold variography and kriging neighbourhood were used instead.

This process allows the model to indicate that, for example, sulphur testing has been conducted in a certain area, but not a quantification of the amount of sulphur present. Figure 13-5 to Figure 13-10 show the distribution of the testing of these variables compared to the main vein, minimum vein and 0.75 g/t Au halo wireframes which were used as mineralized domains for estimation. The estimation of these variables will allow a more targeted approach to the next phase of metallurgical testing, and give an indication of how the geometallurgical variables correlate to the existing lithology and alteration spatially.

For Figure 13-5 to Figure 13-10 below the pink wireframe is the main vein, red wireframe the minimum vein and the orange wireframe the 0.75g/t halo wireframe. All views are long section facing south-east.

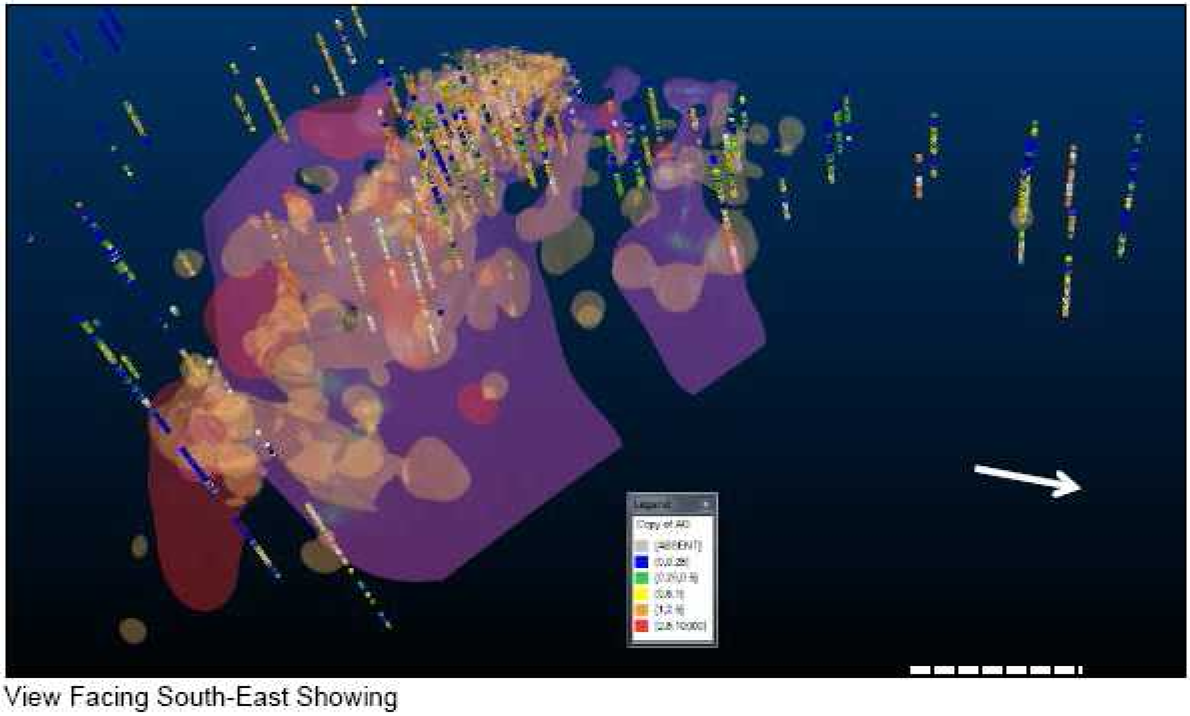


Figure 13-5: Long section showing Silver Data Compared to Mineralized Domains1

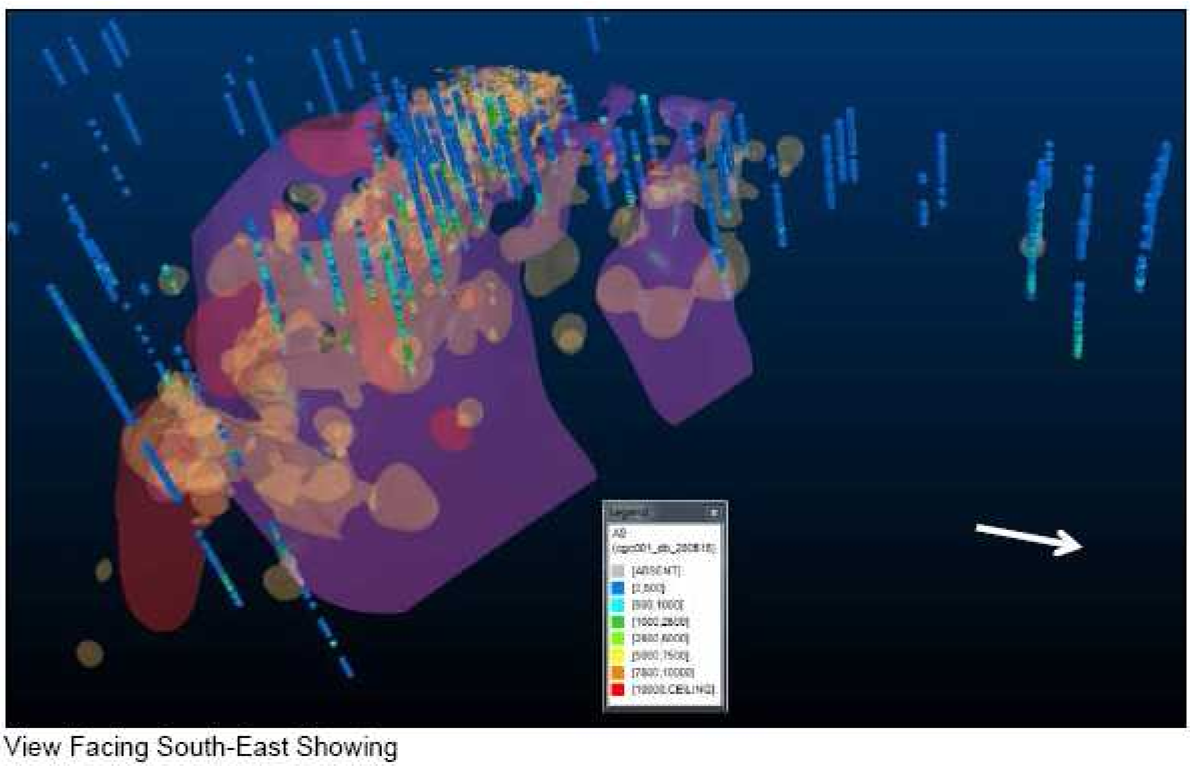
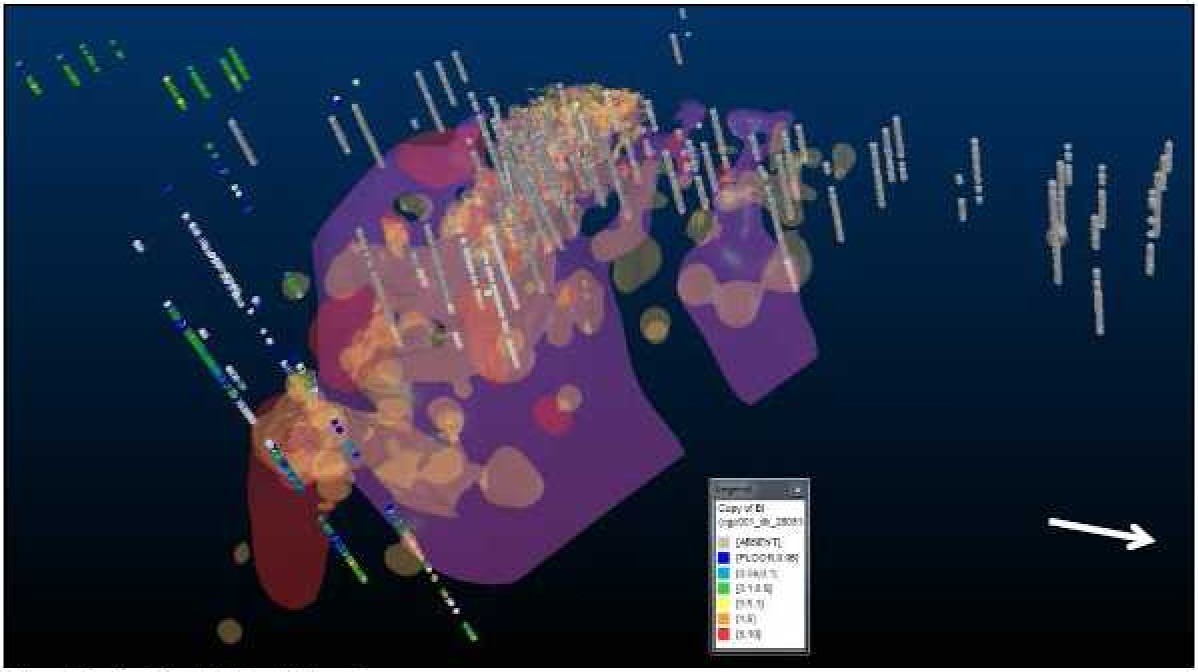
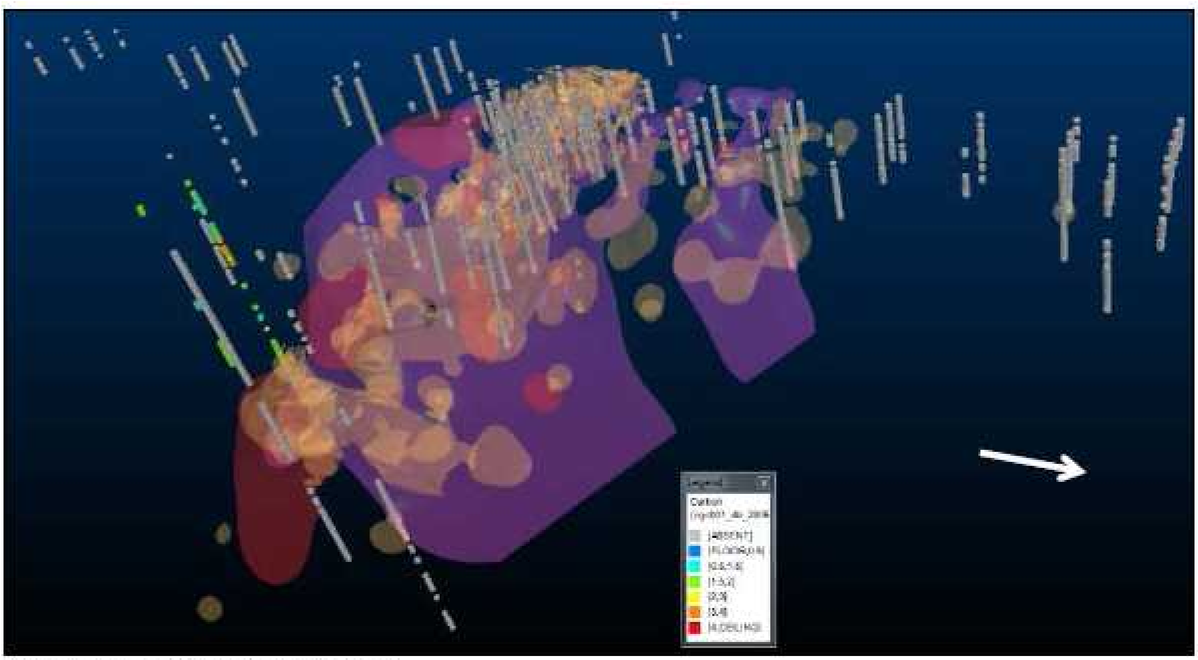


Figure 13-6: Long section showing Arsenic Data Compared to Mineralized Domains



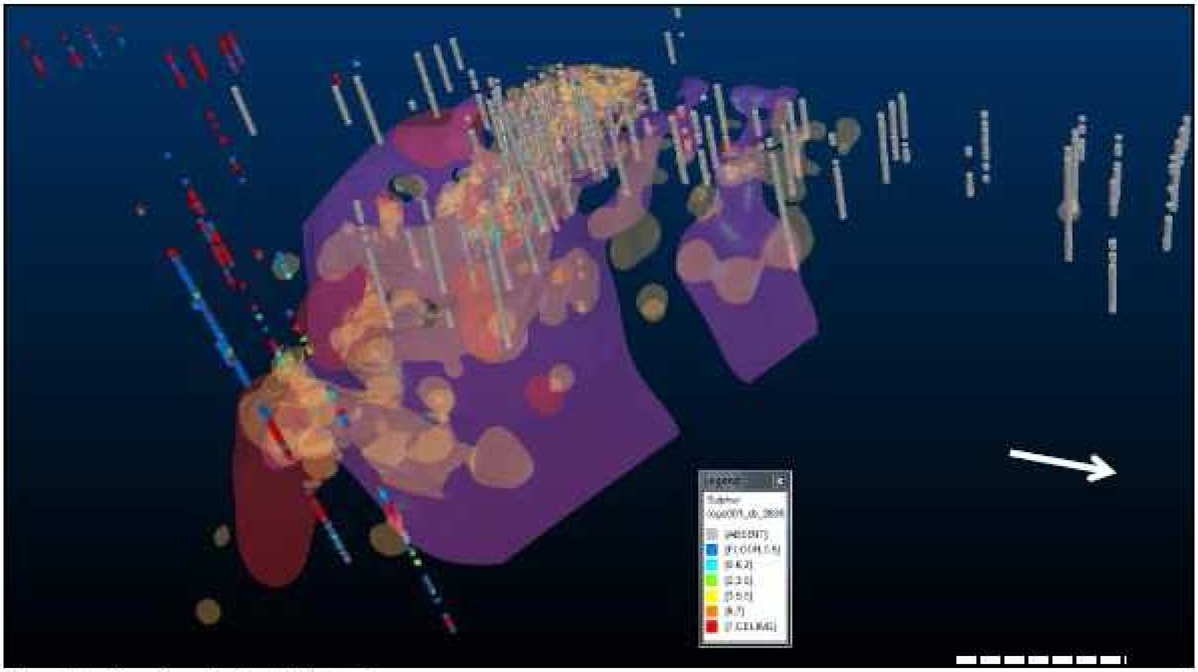
View Facing South-East Showing

Figure 13-7: Long Section showing Bismuth Data Compared to Mineralized Domains



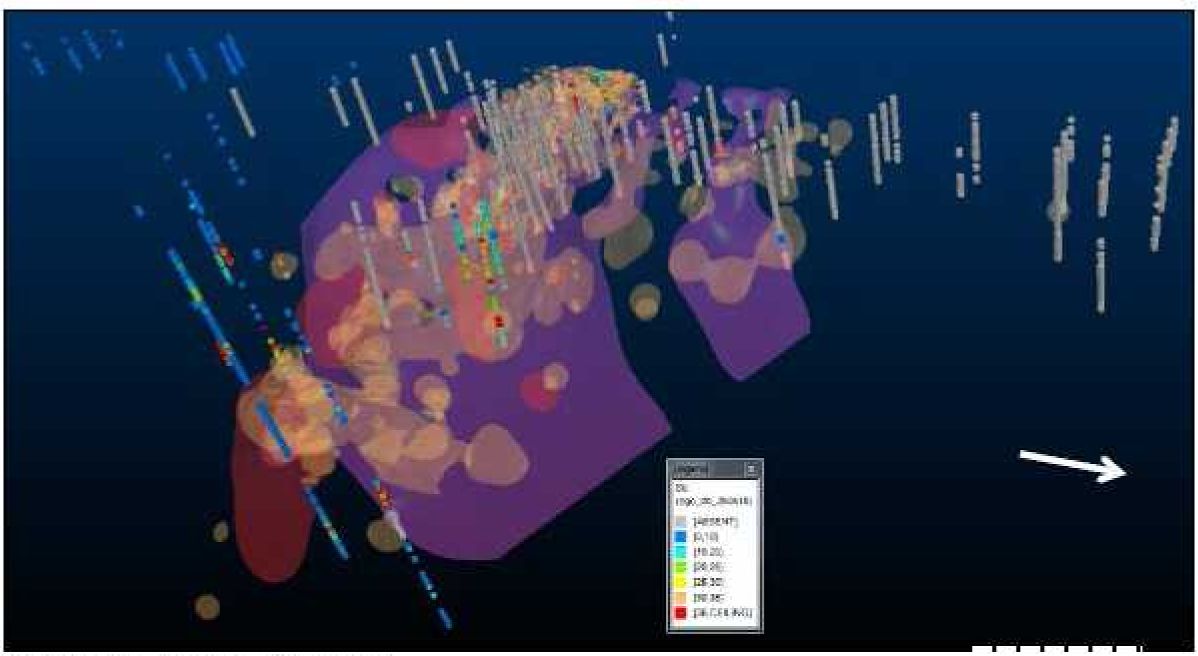
View Facing South-East Showing

Figure 13-8: Long Section showing Carbon, Carbonate Carbon, CO_3, Organic Carbon and1 Total Carbon Data Compared to Mineralized Domains



View Facing South-East Showing

Figure 13-9: Long Section showing Sulphide Sulphur, Sulphate Sulphur and Total Sulphur Data Compared to Mineralized Domains



View Facing South-East Showing

Figure 13-10: Long Section showing Antimony Data Compared to Mineralized Domains

Table 13-9: Historical Metallurgical Reports Used to Construct Geometallurgical Database

Year	Report Name	Company	Number of Samples Tested	Composites	Element Testing	Metallurgical Testing
2005	Maud Creek Flotation Testwork on Maud Creek Sample TMCD04002	Terra Gold	16	16	Au As S Fe	Cyanide recovery
2003	Maud Creek Scoping Testwork on Maud Creek Gold Ore Samples	Harmony Gold	262	14	Au Ag As Cu Pb Zn Hg S Bi Fe Organic C Total C	Cyanide recovery Preg robbing Gravity
1998	Metallurgical Evaluation of the Maud Creek Project	Kilkenny Gold	125	125	As S Fe Organic C CO3	Cyanide recovery Inferred pyrite
1998	Metallurgical Testing of Variability Sample VF9	Kilkenny Gold	89	89		Cyanide recovery
1998	Kilkenny Gold Resources Leach testing - hidden in report #68_116459 Maud Creek, NT, Assays, Lab files, 1996-2005	Kilkenny Gold	21	21		Cyanide recovery Au head and tails
1996	Interim Working Report on the Metallurgical Evaluation of the Maud Creek Gold	Kilkenny Gold	44	23	Au Ag As Cu	Bottle roll test Product size Cyanide recovery

	Project				Fe Organic C Carbonate Total C Sulphate S Sulphide S Total S	Cyanide soluble head and residue grade Reagent data Lime consumption Leach kinetics
1996	Maud Creek-Metallurgy Reports	Kalmet Resources	12	12	Au Ag As Cu Fe	

Year	Report Name	Company	Number of Samples Tested	Composites	Element Testing	Metallurgical Testing
					Organic C Total C Sulphate S Sulphide S Total S	
1995	Maud Creek-Metallurgy Reports	Kalmet Resources	10	10	As S Fe	
1994	Maud Creek Metallurgy AMDEL 2994_OCR	Kalmet Resources	99	99	Au Ag As Cu Sb Sulphide S Total S	Gravity Cyanide recovery
1994	Stage 1 Metallurgical Testing Maude Creek	Kalmet Resources	25	25	Au Ag As Cu Sb Pt Pd R Rh Os Ir Organic C Carbonate C	Gravity Inferred pyrite Cyanide recovery

14 Mineral Resource Estimate

14.1 Introduction

The elements to be estimated are Gold (Au), Arsenic (As), Carbon (C), Copper (Cu), Sulphur (S) and Antimony (Sb). The estimation of the elements has been based on assays sourced from drilling data and metallurgical tests where available. The data available as at March 2016 consisted of reverse circulation (RC), diamond core (DD) and rotary air blast (RAB). RAB samples were excluded from the estimation at the exception of 2 samples.

14.2 Lithology and Structural Model

The Maud Creek lithological model was constructed on the local grid coordinates covering dimensions 1,185,000 m (east) and 1,600,000 m (north). The model incorporates several datasets including Diamond and RAB drilling, AngloGold pit mapping and historic SRK aeromagnetic interpretations (SRK, 1998). All datasets were imported into Leapfrog for subsequent 3D modelling. A topography surface was constructed from collar points and used to constrain the top of the lithological model.

The Maud Creek deposit is hosted within the Proterozoic El Sherana Group units and the mineralization hosted at the faulted contact between the Dorothy Volcanic Member and sediments of the Tollis Formation. The Dorothy Volcanic Member strikes approximately north-south and consists of volcanic tuff with minor interbedded zones of sediments. The Tollis Formation strikes north-south and consists of sandstone and metasediments. The deposit is bound to the east by the Maud Creek Dolerite which intrudes the Tuff sequence. A small Andesite body is also observed to the north of the Maud Creek Open Pit. It is located at the faulted contact between the Sandstone and Tuff, forming a discrete body (Figure 14-1, cover units not included). These key units are also overlain by a thin layer of sedimentary cover and Cambrian Volcanics.

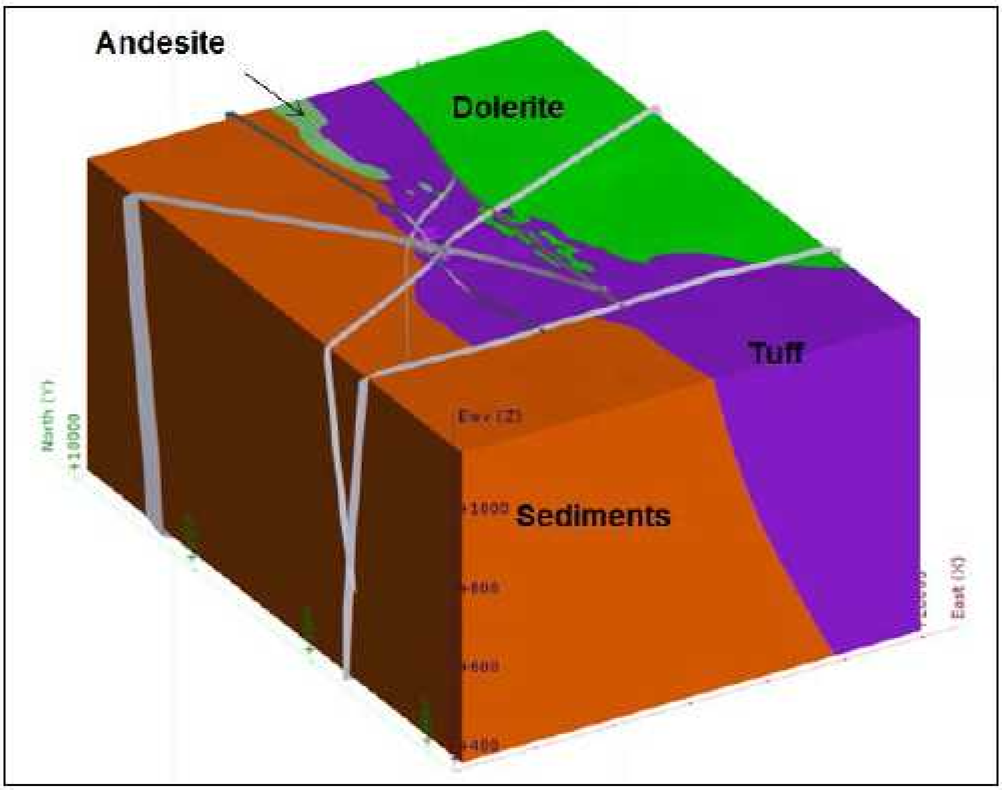


Figure 14-1: Maud Creek Deposit lithology model showing primary units, cover not shown

The key unit formations described above were determined based on the logging code 'Lith Code 1' from the dataset provided by Newmarket Gold. For the purpose of 3D modelling the combination of several lithologies was sometimes required to form a key lithology group (Table 14-1).

Table 14-1: Maud Creek Lithology Model Groupings

Lithology Group	**Codes**
Cover	ALUV, CLA, CLAY, CLY, GO, LOM, MUD, SOIL, SPLT, SND, CALC, CLCR, LMST, BLT
Dolerite	DLT, DOL, INTD
Tuff	IGM,MAFT,TUF
Sediments (Sandstone)	MSED, GYWK, QTZ, Slst, MDST, SDST, Ssl
Andesite	ANT
Vein	BX, VEBX, VEIN, SHLE

Within the model area eight faults have been identified (Figure 14-2) based on historic AngloGold pit mapping as well as aeromagnetic interpretations conducted by SRK Consulting (SRK, 1998). Orientations of these structures were extracted from the mapped pit data and interpreted based on available datasets. Generally, the faults exhibit reverse movement, with limited offsets in the range of meters. Figure 14-3 shows the interpreted fault architecture, indicating apparent reverse movement along faults. To the south of the Maud Creek Deposit a major east-west structure with sinistral strike-slip movement has been interpreted based on the drilling data and aeromagnetic interpretations. Additional faults are likely present in the modelled area, however only faults which have significant structural control on the deposit have been constructed.

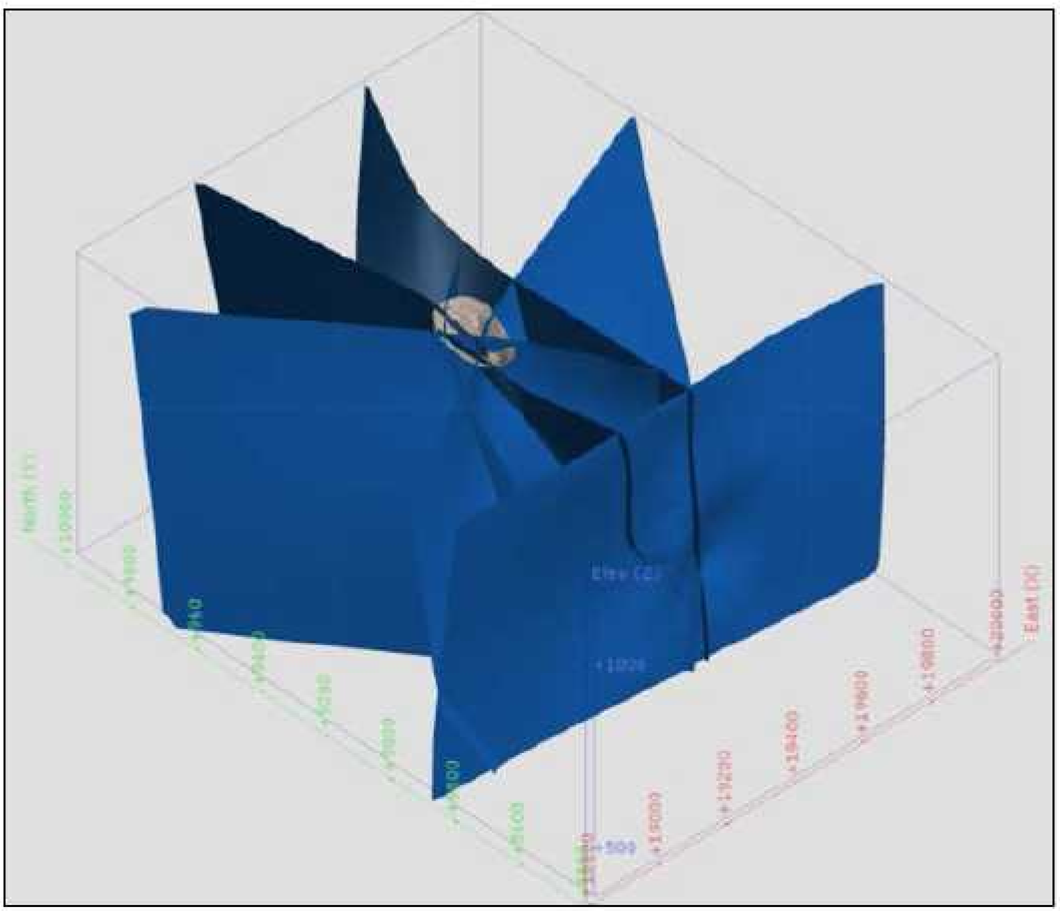


Figure 14-2: Interpreted fault architecture of the Maud Creek Deposit

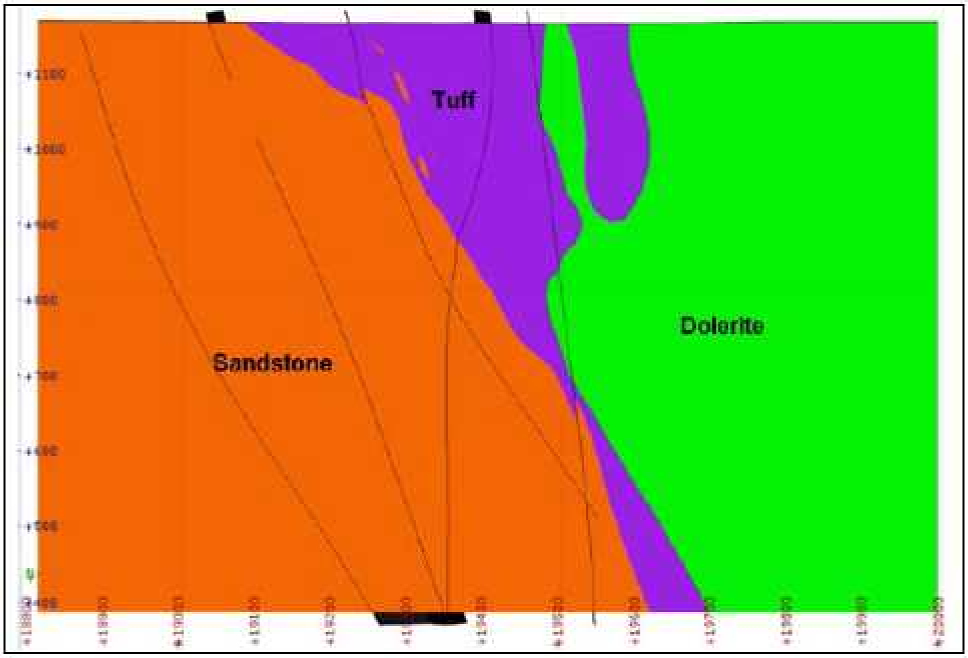


Figure 14-3: Cross section view looking north (northing 9008mN)

14.3 Vein Model

Three veins have been interpreted based on the main lithology logging (Lith Code 1); a continuous primary vein, discrete upper vein and discrete lower vein (Figure 14-4). The veins generally follow the main faulted contact between the Sandstone and Tuff units and have an apparent plunge to the south-east, which follows the contact fault (Figure 14-5). The primary vein typically hosts the highest gold grades and lies within the Sandstone/Tuff contact, however deviations from this contact are evident. The upper and lower veins generally hosts lower gold grades and exhibit limited continuity located above and below the faulted contact. Additional vein material was evident in the logging however these veins typically have distinctly limited continuity and have not been included within the final model. The presence of contact veining decreases to the north of the Anglo Pit, based on available drilling.

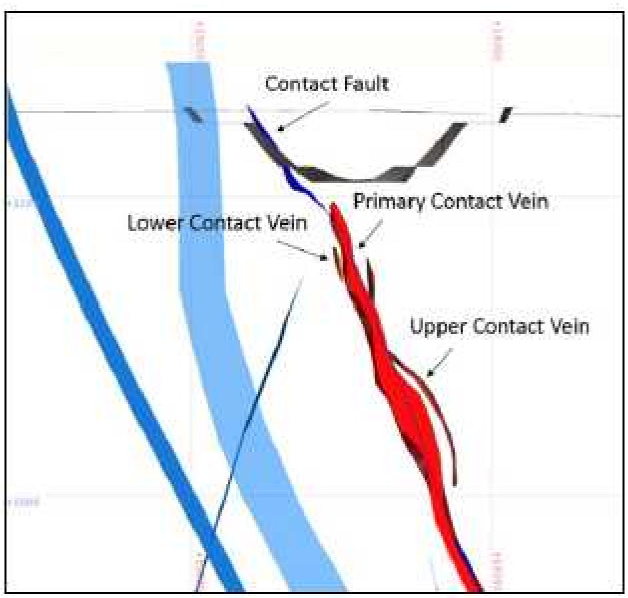


Figure 14-4: Cross section looking north of the modelled veins (primary, upper and lower)

Note: Contact fault (dark blue) and two of the eight faults modelled (light blue)

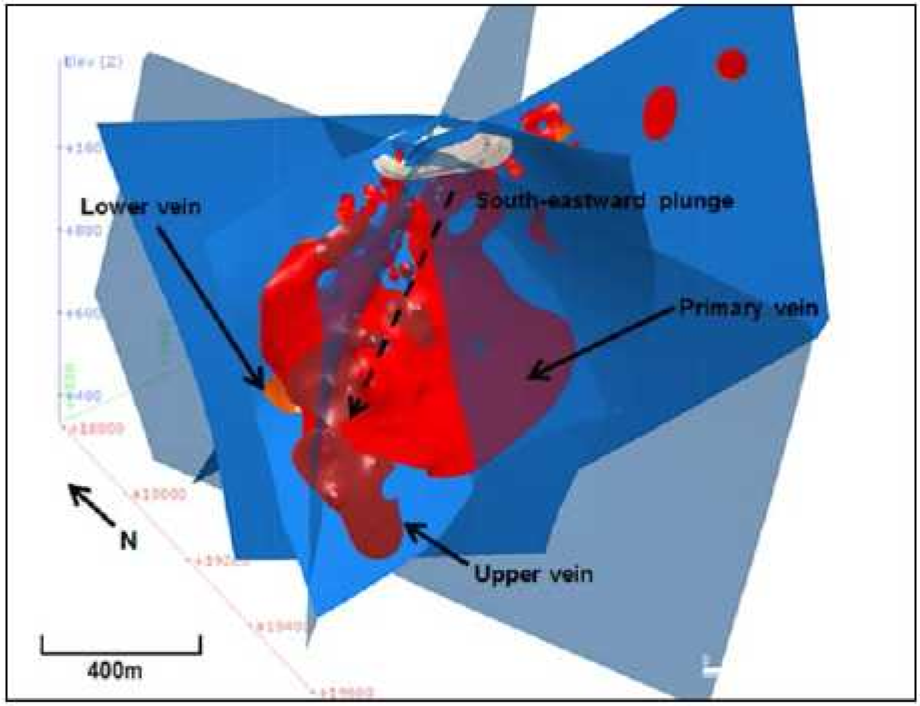


Figure 14-5: Vein architecture of the Maud Creek Deposit illustrating three primary veins and indicating a south-eastward plunge

The primary vein is strongly associated with the Sediment/Tuff contact but does not strictly follow the contact. Therefore, it has been modelled as an 'overprinting' volume on the sediment, tuff, dolerite and andesite lithology wireframes. Lithological codes BX, VEBX, VEIN or SHLE are present in most holes that intersect the contact. Where these codes are absent the vein has, in most cases, been modelled as pinched out. Where SHLE is present at, or proximal to, the contact, the intercept will carry grades similar to that of the BX, VEBX and VEIN intercepts (approximately 4 g/t Au). There are also intervals of SHLE located distal to the contact, and these do not carry grade.

For estimation purposes the upper and lower veins were combined, resulting in two domains; main vein (primary) and minor vein (upper and lower).

14.4 Grade Halo Models

Outside of and adjacent to these lithologically defined veins are many intercepts which carry similar grades to those within the vein itself. These extend up to 25 m into the hanging wall and to a lesser extent into the footwall. In addition, a greater than 0.1 g/t Au halo can be observed up to 50 m into the hanging wall and occasionally in the footwall. Excluding the veins mentioned above, the interval statistics did not indicate any grade distinction with lithology type.

Mineralization consists of two distinct zones (east and west) controlled by two north-south striking structures (Maud Creek Contact Fault and North-South Fault 1) (Figure 14-6). The western zone of the mineralization is primarily controlled by the structural contact between the footwall Sandstone and hanging wall Tuff. This zone illustrates the strongest concentration and highest grade of mineralization within the Maud Creek deposit. The fault contact strikes approximately north-south and is interpreted to have undergone reverse movement. The fault structure is filled with quartz stockwork veins; the primary host of high gold mineralization, with additional gold hosted within the surrounding wall rock. The eastern zone of mineralization is controlled by a north-south striking structure that has been inferred based on aeromagnetic interpretations (SRK, 1998) (Figure 14-6). This structure lies proximal to the contact between the Maud Creek Dolerite and Tuff units. Within this zone gold generally forms steeply dipping discrete lenses with limited continuity noted along strike.

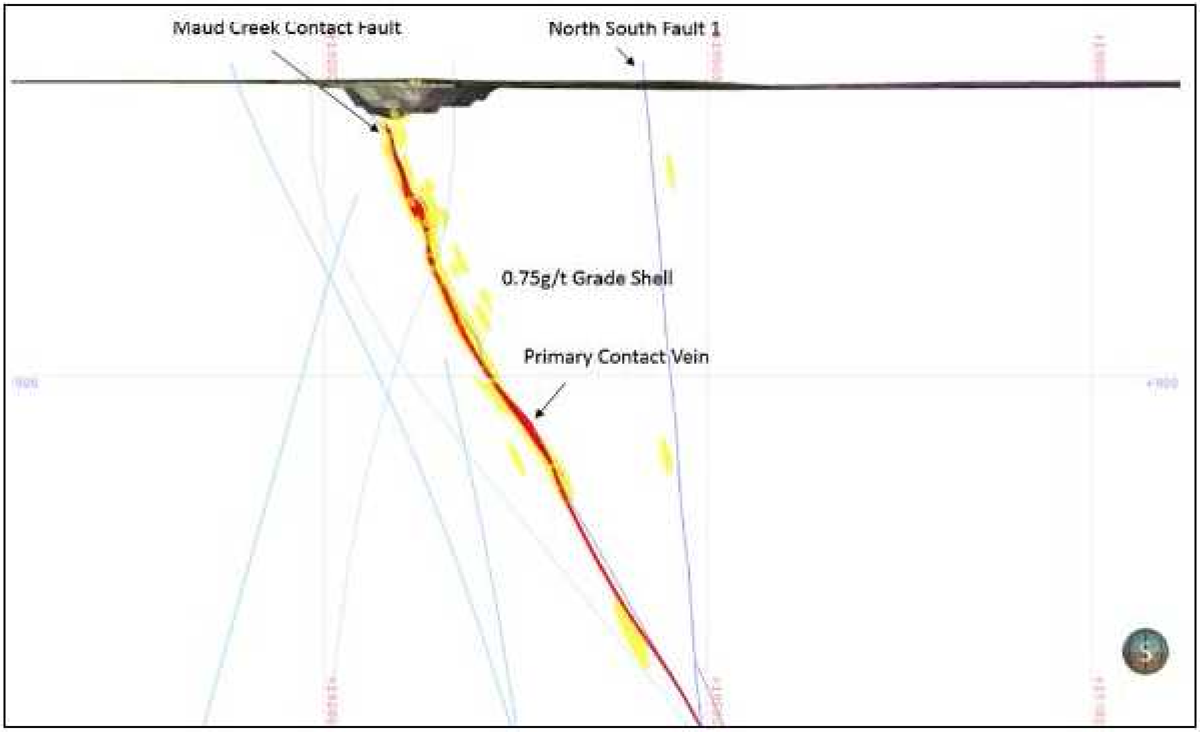


Figure 14-6: Cross section illustrating 0.75 g/t grade shell (yellow), primary contact vein (red) and fault architecture (blue)

To capture the complexity of the interactions and multiple orientations of the many faults within the deposit, grade shells generated in Leapfrog were used to model the wall rock mineralization in two stages. Although not obvious in the statistics, observation of the grade downhole suggested a sharp break at around 0.75 g/t Au. Therefore, two nested grade shells were modelled at 0.75 g/t Au and 0.1 g/t Au to be used as estimation domains. These were generated using grade from all intercepts, including those within the vein model.

In some areas of widely spaced drilling (50 m - 100m down dip) the grade shell models could not be made continuous, even though the vein had been interpreted as continuous. This is a limitation of the Leapfrog software and chosen methodology. The alternative was to manually wireframe this domain but this was considered more time consuming and less likely to capture the multiple orientations observed. There are also locations where the grade observed at the contact was too low or thin to sustain a grade shell (Figure 14-7).

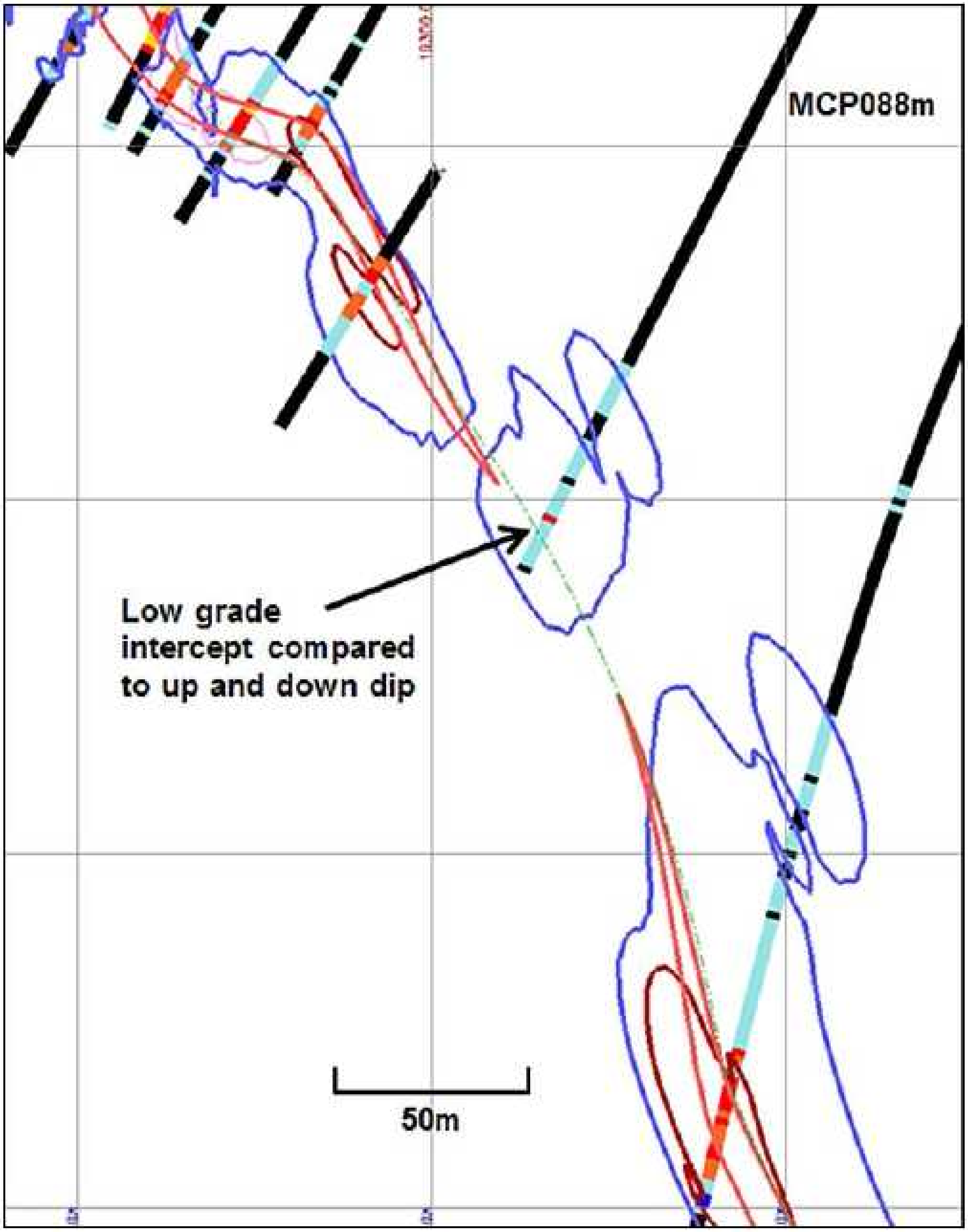


Figure 14-7: Cross section looking north (9250mN) exhibiting low grade hole MCP088 through the fault contact

14.4.1 Assumptions on non-continuity of grade adjacent to vein

The vein material (VEIN, VBX, BX or SHLE) is observed in the majority of holes that go through the contact between the Tuff and the Sediments and the model assumes Vein continuity between holes where the vein lithologies are recorded. The mineralization above 0.75 g/t Au in the footwall (Sediments) and hanging wall (Tuff) directly adjacent to the vein does not show similar continuity. Mineralization > 0.75 g/t Au may be present or absent in either the footwall or hanging wall from one hole to the next. This is observed throughout the deposit. An example is shown in Figure 14-8. MCP125 contains almost no grade in the footwall but 5 m of moderate grades in the hanging wall. The next hole down dip, MD024, contains 7 m of moderate grades in the footwall and 4 m of moderate grade in the hanging wall. The next hole down dip, MCP469 contains 2 m of grade in the footwall and 4 m of grade in the hanging wall. An assumption of continuity from hole to hole for footwall and hanging wall material, particularly in the footwall, cannot be made and this is reflected in the limited connectivity of the 0.75 g/t and 0.1 g/t Leapfrog grade halo domains.

It is also worth noting that the highest grades do not always occur in the vein and that some vein material is very low grade (Figure 14-9).

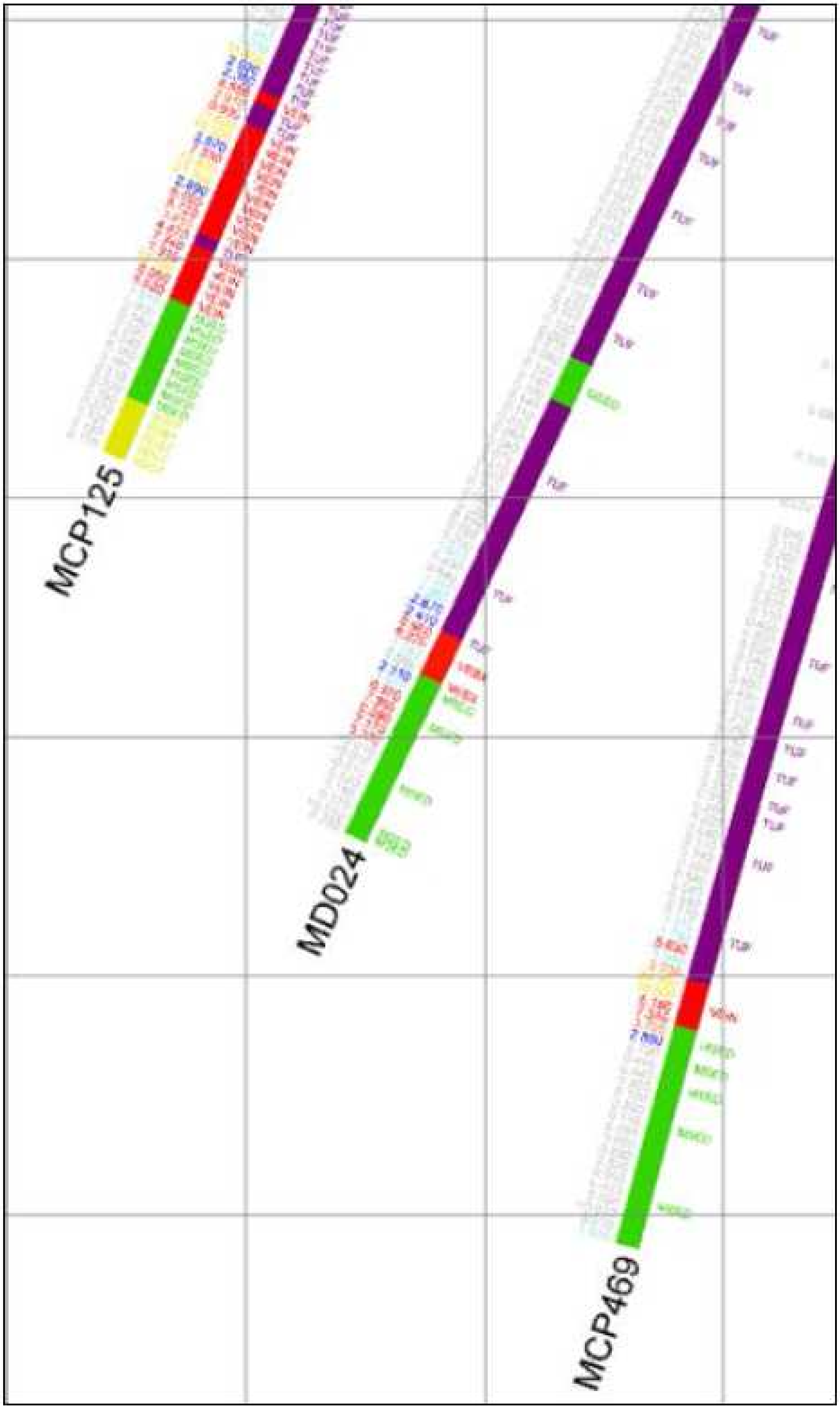


Figure 14-8: Section 9150N (20 m grid)

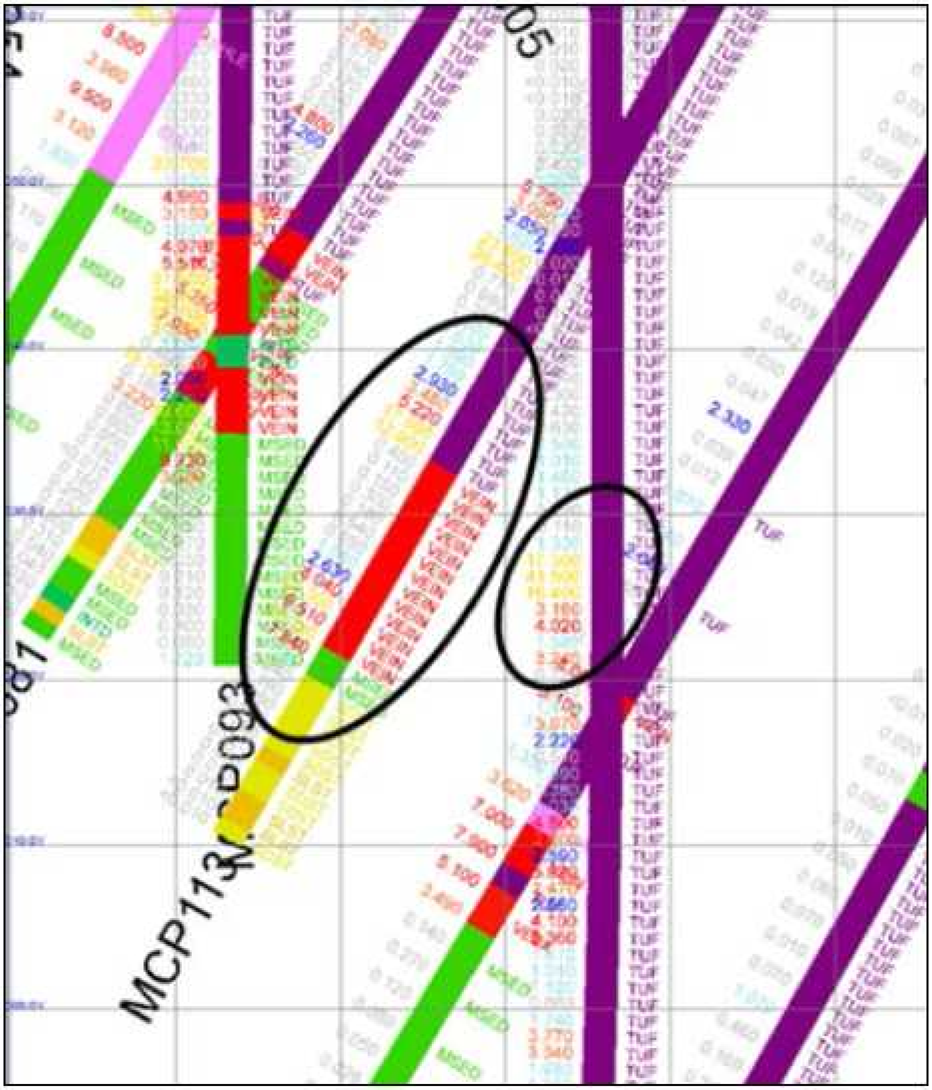


Figure 14-9: Section 9175 (20 m grid)

14.5 Domaining

The Mineral Resource was estimated into six domains as described in the previous sections:

- 0P1WEST;
- 0P75WEST;
- 0P1EAST;
- 0P75EAST;
- MAIN (major vein); and
- MINOR (minor veins).

14.6 Compositing

GEOVIA GEMS software was used to desurvey and composite the drilling. Drillholes were flagged by the different domains and then composited to 1 m intervals within the units with a minimum length of 0.01 m (Figure 14-10). This resulted in 16.5% of the composites with a length less than 1 m, however the influence of small intervals on the gold grade is limited (Table 14-2). A total of 22,449 composite samples were created.

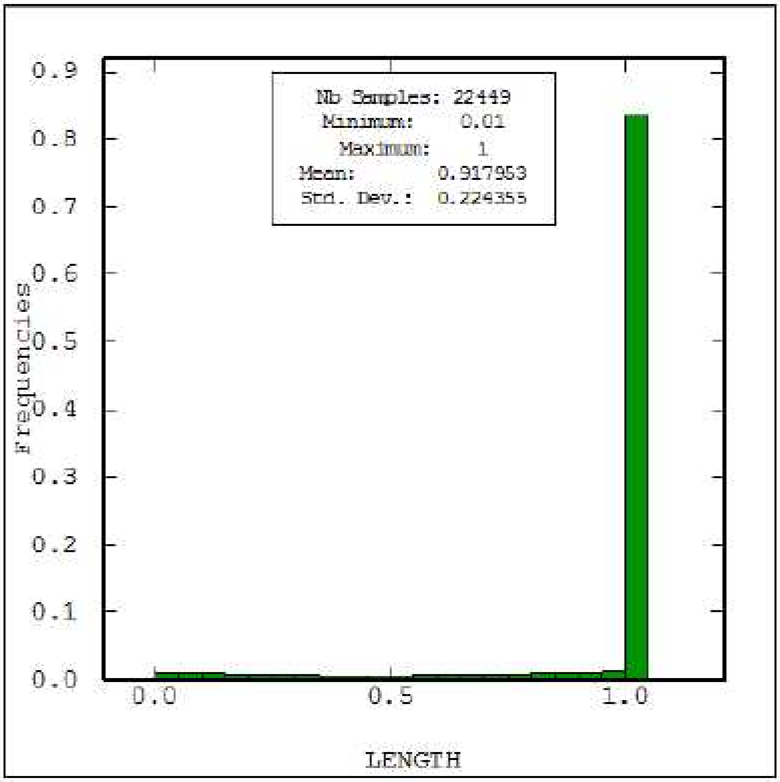


Figure 14-10: Composite length histogram

Table 14-2: Influence of the composite length on gold mean grade

Domain	Au mean grade (g/t)	Length weighted Au mean grade (g/t)	Differences
0P1WEST	0.54	0.53	1.7%
0P75WEST	4.61	4.75	-3.0%
0P1EAST	0.64	0.64	-0.6%
0P75EAST	3.71	3.86	-4.0%
MAIN	6.50	6.54	-0.5%
MINOR	5.72	6.01	-5.0%

14.7 Metallurgical Samples

The secondary elements arsenic, carbon and sulphur in the database included combined assays gathered from metallurgical studies and drill assays. The numbers of assays from metallurgical studies were minimal. Due to the compositing length the values tend to be repeated several times along a same hole creating a bias in domains with limited samples. Arsenic is an important element and the influence of the arsenic assays from metallurgical studies was analyzed. Table 14-3 indicates a fairly strong similarity between the Arsenic grades from drilling assays and metallurgical samples. Only domains with lower sample numbers indicate strong dissimilarity. Quantile-Quantile plot were also used to assess any arsenic distribution differences. The distributions were negligibly influenced by the additional metallurgical samples as shown in the MAIN domain in Figure 14-11.

Table 14-3: Comparison of arsenic assay from drilling only and including metallurgy samples

Domain	Samples Number			AS mean grade (g/t)		
	Drilling	Metallurgical	Difference	Drilling	Metallurgical	Difference
0P1WEST	5271	5365	1.8%	571	600	4.9%
0P75WEST	2149	2440	11.9%	2493	2630	5.2%
0P1EAST	1182	1188	0.5%	791	820	3.5%
0P75EAST	42	102	58.8%	3524	8311	57.6%
MAIN	641	719	10.8%	3352	3442	2.6%
MINOR	98	147	33.3%	2867	3526	18.7%

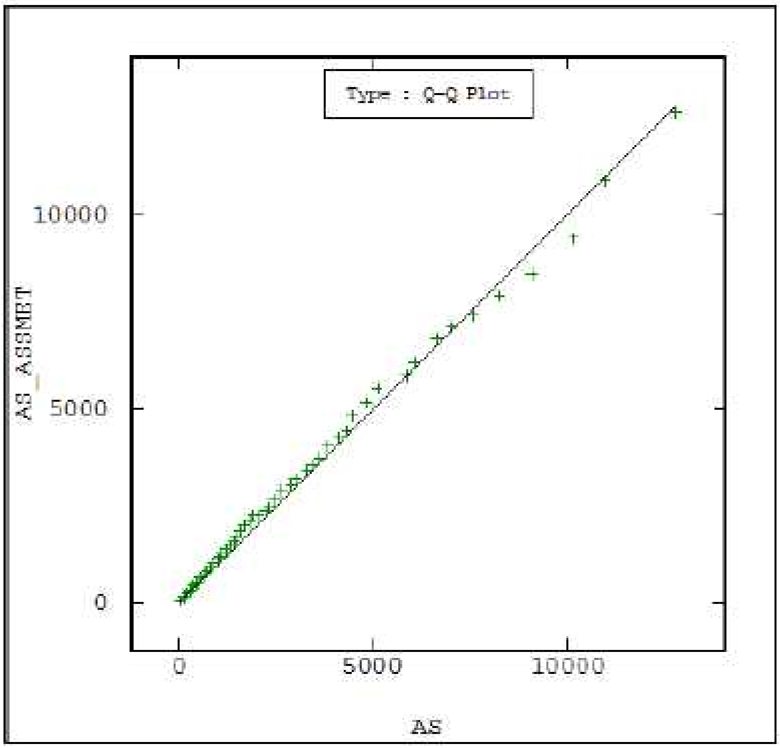


Figure 14-11: MAIN – Quantile-Quantile plot comparing arsenic from drilling only and including metallurgy samples (ASSMET)

In MINOR and 0P75EAST, the assays from the metallurgy study represent 33% and 58.8%, respectively, of the total samples. Most of these samples are located on the same hole and have been duplicated due to the compositing, as shown for the MINOR domain in Figure 14-12. Despite the differences observed in Arsenic grades the 0p75East domain and the MINOR domain all metallurgical samples were included in the drilling dataset and used for the estimation as the spatial location locations of the met samples tend to be different to the regular sample grades.

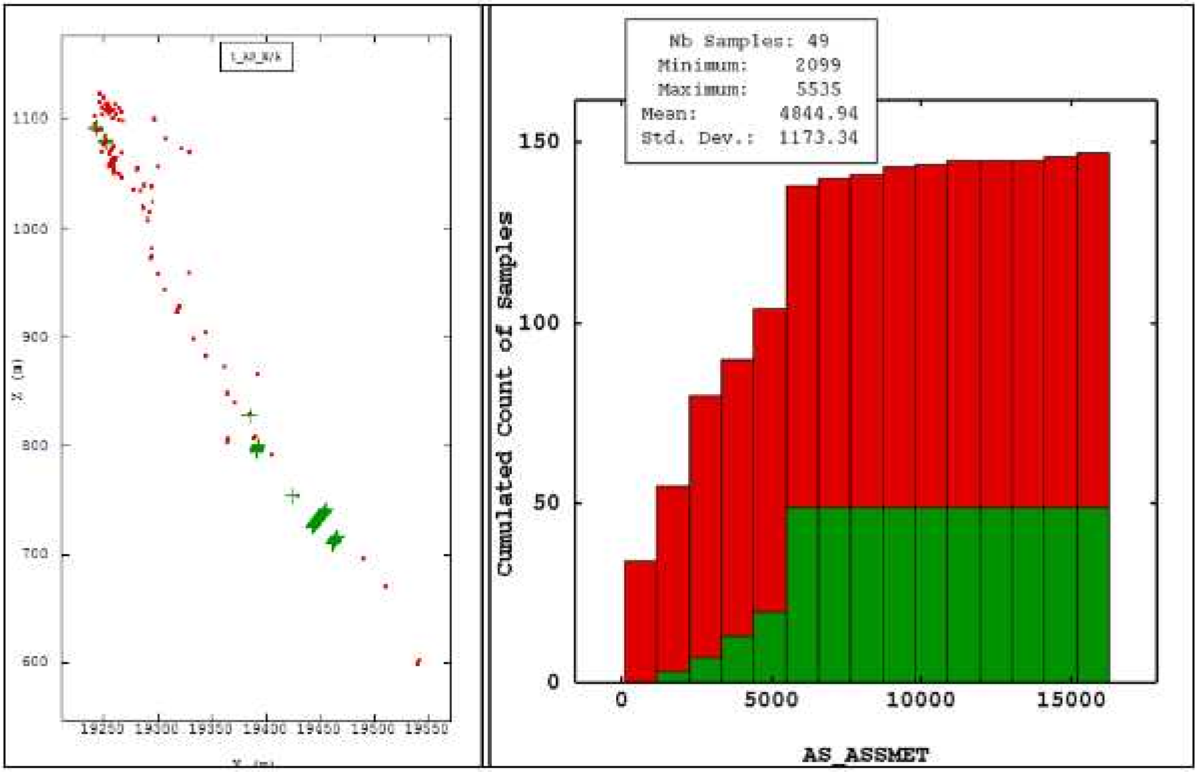


Figure 14-12: MINOR – Arsenic - Left: Plan view of data, Right: Cumulative histogram, Red: Arsenic from drilling only, Green Arsenic from the metallurgical samples

14.8 Block Model Definition

The block model size for estimation was set to 5 m x 10 m x 10 m (X, Y, Z) as this was a reasonable selective mining unit size for both open pit and underground studies, was proportional with the steep dipping north striking orientation and was a reasonable compromise to fit both the close and wiser spaced drilling.

The block model for the estimation is a proportional block model based on the domain boundaries. A sub-block model of 0.625 m x 1.25 m x 1.25 m (X, Y, Z) created in Geovia Surpac was used to visualise the blocks report the resources. The vein models are quite narrow in places and this level of sub blocking was the largest that reproduced the actual wireframe volumes within acceptable limits.

Details of the model are shown in Table 14-4.

Table 14-4: Block model properties

	X	Y	Z
Origin (m) (Lower SW corner)	19,000	8,645	400
Cell size (m)	5	10	10
Number of cells	140	140	80
Sub cell size	0.625	1.25	1.25

14.9 Grade Interpolations

The estimation strategy differed depending on the elements and domains. The elements that were estimated are Au, As, Ag, C, Cu, Sb and S. No strong correlations between the elements have been noted.

The Ordinary Kriging (OK) method of interpolation was selected to estimate the gold and arsenic grade within all six domains as these had the most complete coverage. The other elements were separately estimated using an Inverse Distance of order 2 (ID2) within the western domains and MAIN domain and a Nearest Neighbourhood (NN) method of interpolation for the eastern domains and MINOR domain. Ag, C, Cu, Sb, and S do not have complete sampling coverage and as a consequence the block estimates also do not have complete coverage of the Au estimated blocks.

The methodology used for the estimation consisted of:

Performing cell declustering tests;
Studying the influence of outliers;
Validating the choice of using samples from all drillhole type;
Studying the spatial variability for gold and arsenic (variography);
Defining an estimation strategy; and
Validating the estimation.

14.10 Declustering

Drilling includes various types of holes with unequal spacing, the highest drilling density corresponding to the open pit area and near surface. A cell declustering test was performed using a grid specific range specific to each domain and element. The declustering results are shown in Table 14-5, Table 14-6 and Table 14-7. A positive difference means that the declustered grade is lower than the undeclustered grade.

Table 14-5: Declustering results in the vein domains

Elements	MAIN			MINOR		
	Cell Size (m)	Declustered Mean (g/t)	Differences	Cell Size (m)	Declustered Mean (g/t)	Differences
AU	40x40x40	5.67	13%	90x90x90	3.26	43%
AS	90x90x90	2,993.1	13%	90x90x90	2,707.9	23%
AG	60x60x60	3.42	-0.8%	50x50x50	2.84	12.5%
CU	60x60x60	176.3	2.9%	50x50x50	88.9	29.9%
C	60x60x60	1.62	1.4%	50x50x50	1.04	-2.6%
SB	60x60x60	19.4	14.4%	50x50x50	29.95	18.1%
S	60x60x60	1.43	-3.1%	50x50x50	1.8	-5.1%

Table 14-6: Declustering results in the western domains

Elements	0P1WEST			0P75WEST		
	Cell Size (m)	Declustered Mean (g/t)	Differences	Cell Size (m)	Declustered Mean (g/t)	Differences
AU	40x40x40	0.47	13%	40x40x40	3.11	32%
AS	50x50x50	817.8	-36%	30x30x30	2,564.2	3%
AG	60x60x60	1.79	13.5%	60x60x60	2.99	28.2%
CU	60x60x60	91.2	4.2%	60x60x60	112.4	5.5%
C	60x60x60	1.75	13.3%	60x60x60	1.59	0.4%
SB	60x60x60	20.0	14.8%	60x60x60	26.2	12.6%
S	60x60x60	1.07	-21.1%	60x60x60	1.32	1.5%

Table 14-7: Declustering results in the eastern domains

Elements	0P1EAST			0P75EAST		
	Cell Size (m)	Declustered Mean (g/t)	Differences	Cell Size (m)	Declustered Mean (g/t)	Differences
AU	40x40x40	0.55	14%	40x40x40	3.77	-2%
AS	40x40x40	991.2	-21%	50x50x50	1,0024.8	-21%
AG	30x30x30	0.91	-4.5%	30x30x30	8.23	0.0%
CU	30x30x30	60.1	2.8%	30x30x30	91.6	0.0%
C	30x30x30	1.89	-0.2%	30x30x30	-	-
SB	30x30x30	33.3	3.1%	30x30x30	62.5	0.0%
S	30x30x30	1.24	-69.6%	30x30x30	1.80	-6.4%

14.11 Outliers

The element grades have a skewed distribution within each domain as shown by the coefficient of variation shown in, Table 14-8 and Table 14-9. The histograms show a long tail for high grades. In the eastern domains and Minor domain, the grade distributions of the elements other than gold and arsenic are not well represented due to a limited number of samples.

MAIN domain, Gold grade distribution is strongly skewed with a coefficient of variation of 5.2. Significant high grade samples are skewing the Gold grade distribution. In particular, two samples have been identified above 200g/t which are relatively isolated from the main distribution and have a strong impact on the statistics and variography (Figure 14-13). They are located approximately 425 m below surface at the edge of the domain (Figure 14-14).

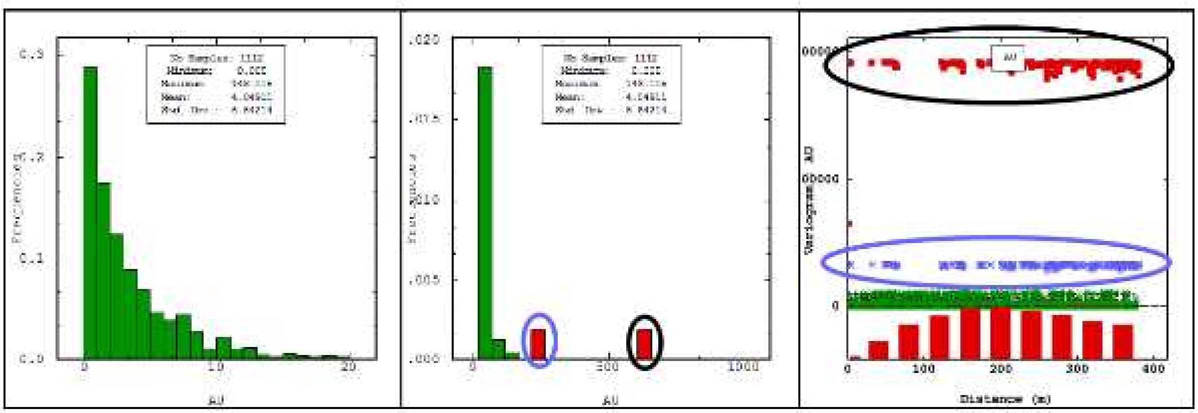


Figure 14-13: MAIN - Left and Middle Au histogram: from 0 - 20g/t and from 20 - 1000 g/t and Right: Variogram cloud, Red and blue: outliers

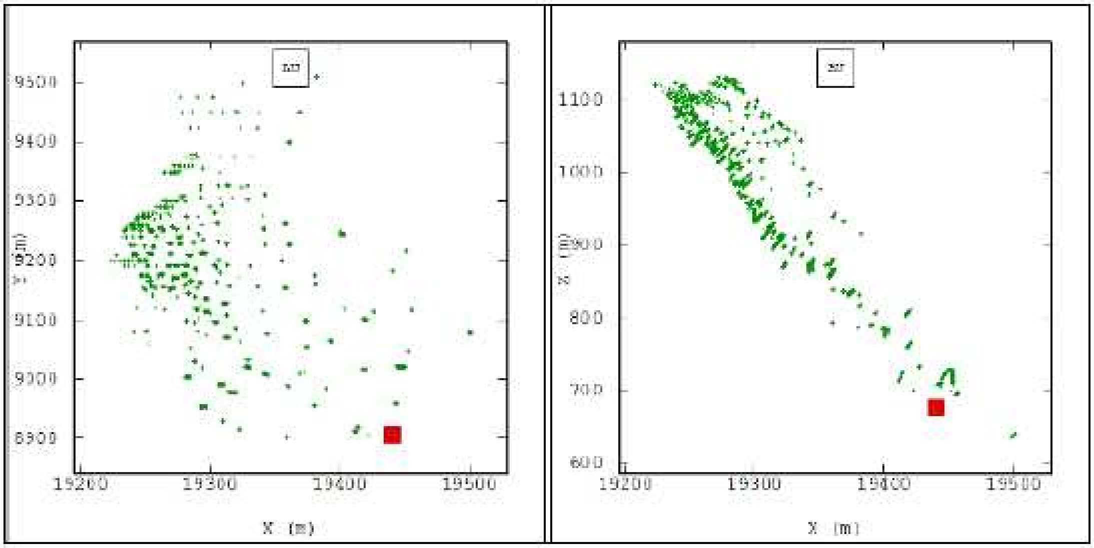


Figure 14-14: MAIN – Au samples location – Left: XoY view, Right: XoZ view
Red: outliers

Further analysis was undertaken for each element within each domain. According to the influence of the outliers on the variography and their locations, a top cut was applied or not. When the outliers have a strong impact, a top cut was applied. Values above the top cut were replaced by the top cut threshold. Table 14-8 and Table 14-9 summarise the treatment of the outliers per domain.

Table 14-8: Outlier treatment in veins domains

Elements	MAIN			MINOR		
	# Samples	Declustered Coeff. Variation	Top cut threshold (g/t)	# Samples	Declustered Coeff. Variation	Top cut threshold (g/t)s
AU	1,114	5.2	50	4,615	1.7	50
AS	719	0.9	-	147	0.8	10,000
AG	147	1.0	-	28	0.8	6.5
CU	148	2.5	-	28	1.0	300
C	43	0.7	2	31	0.2	2
SB	57	1.0	-	9	0.8	50
S	121	0.7	-	47	0.4	2.1

Table 14-9: Outlier treatment in western domains

Elements	0P1WEST			0P75WEST		
	# Samples	Declustered Coeff. Variation	Top cut threshold (g/t)	# Samples	Declustered Coeff. Variation	Top cut threshold (g/t)s
AU	11,091	2.5	50	4,615	1.7	-
AS	5,365	1.7	-	2,440	1.1	-
AG	1,369	2.2	-	499	2.0	-
CU	1,443	2.0	-	505	1.5	-
C	96	0.7	-	229	0.8	-
SB	298	0.8	-	233	0.9	-
S	263	0.9	-	388	0.7	-

Table 14-10: Outlier treatments in eastern domains

Elements	0P1EAST			0P75EAST		
	# Samples	Declustered Coeff. Variation	Top cut threshold (g/t)	# Samples	Declustered Coeff. Variation	Top cut threshold (g/t)s
AU	3,408	3.3	40	386	1.1	-
AS	1,188	2.5	-	102	0.9	12,000
AG	192	1.2	5.5	11	0.7	-
CU	190	1.1	-	11	0.4	-
C	31	0.2	2	0	-	-
SB	73	0.4	-	11	0.4	-
S	46	1.0	1.9	71	0.5	1.9

14.12 Drillhole Types

The drilling dataset used for the estimation consists of 677 drillholes including 1 RAB hole for 1.91 m composite length, 93 DD holes totalling 6107.8 m composite length and 583 RC holes totalling 14,497 m composite length. 256 RC holes are from grade control RC drillholes and are called RCGC. Figure 14-15 shows in red the location of the RCGC on the projected northing section.

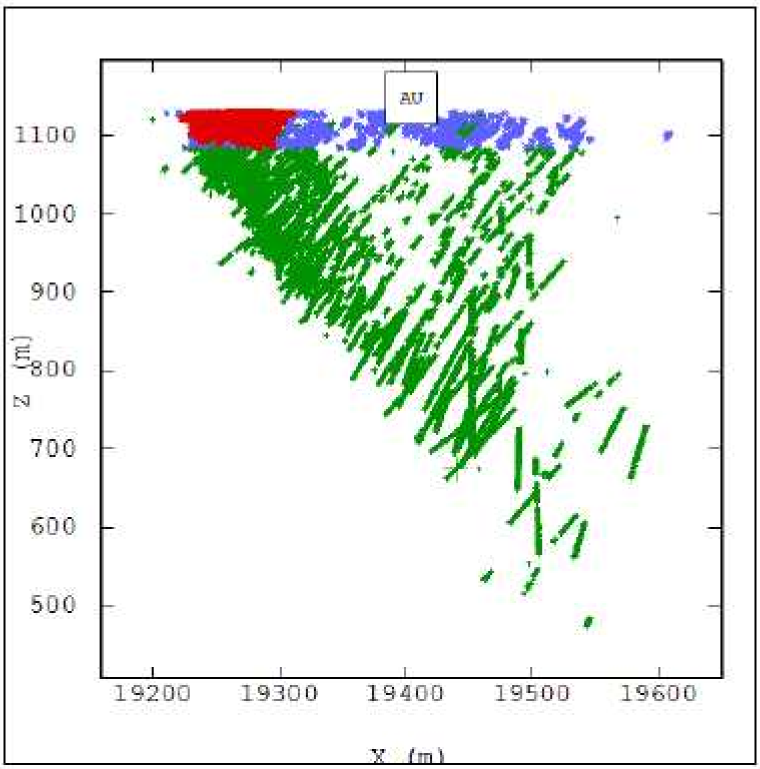


Figure 14-15: AU samples collected in the RC drillholes, Red: RCGC, Blue: RC samples above 1,084 mRL and Green: remaining RC

The statistical comparison of drillhole types was undertaken using weighted composite data and outlier treatment.

Figure 14-16 compares the differences Au mean grade between RC above 1,084 mRL and RCGC. RCGC were not found in the eastern domains. The Au mean grade compares relatively well between both drilling types at the exception of MINOR domain. In MINOR domain, Figure 14-17 indicates that the Au distribution from RC samples above 1,084 mRL is not well defined after 24 g/t compares to the distribution within the RCGC samples. SRK considers that there is no bias between the RC and RCGC samples.

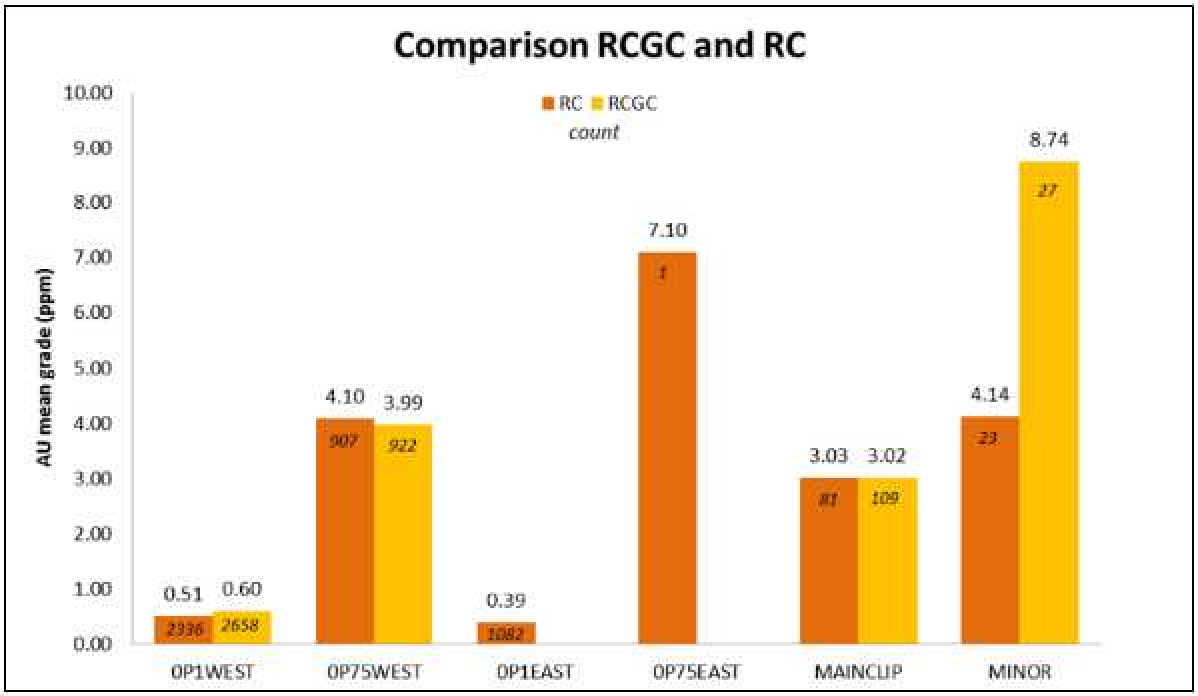


Figure 14-16: Comparison of weighted and top-cut Au mean grade between RC and RCGC above 1084 mRL

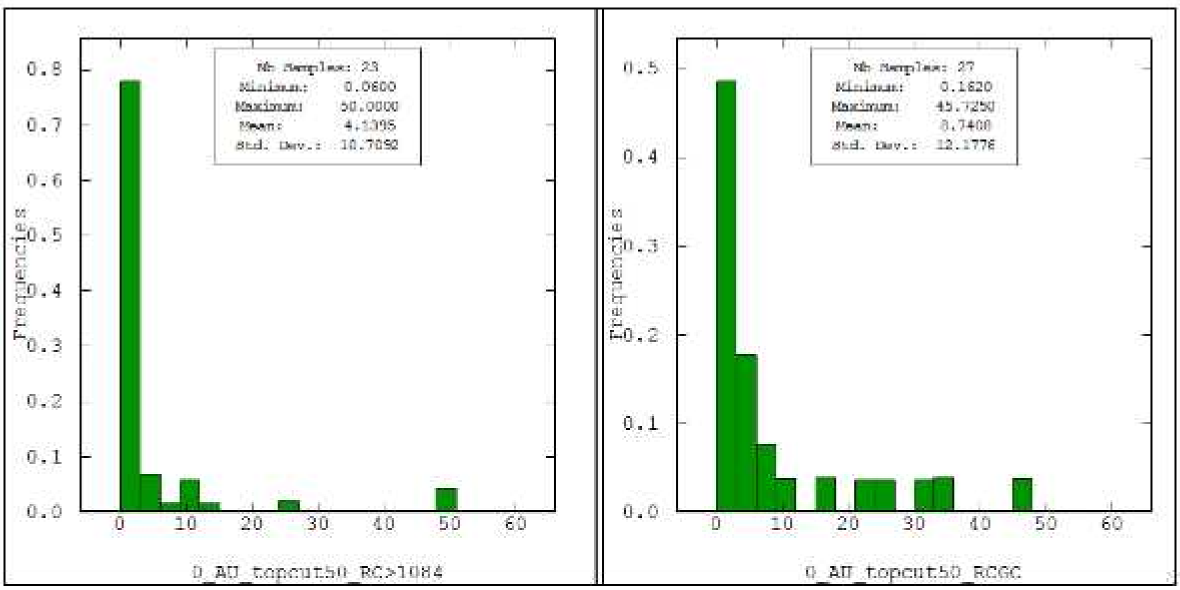


Figure 14-17: MINOR – Au distribution – Left: RC above 1,084 m, Right: RCGC

Samples from RCGC drillhole type were combined with all the samples from RC drillhole type. From now on any reference to RC drillhole type includes RCGC drillhole type.

Figure 14-18 compares the Au mean grade between the DD and RC within each domain. However the differences are less than 10% in most domains, the statistics fluctuate with the number of samples and are influenced by high grade values. Two third of the composite samples are from RC samples. Small domains with low number of samples tend to have more discrepancies between data types. SRK considers that there is no bias between the RC and DD samples.

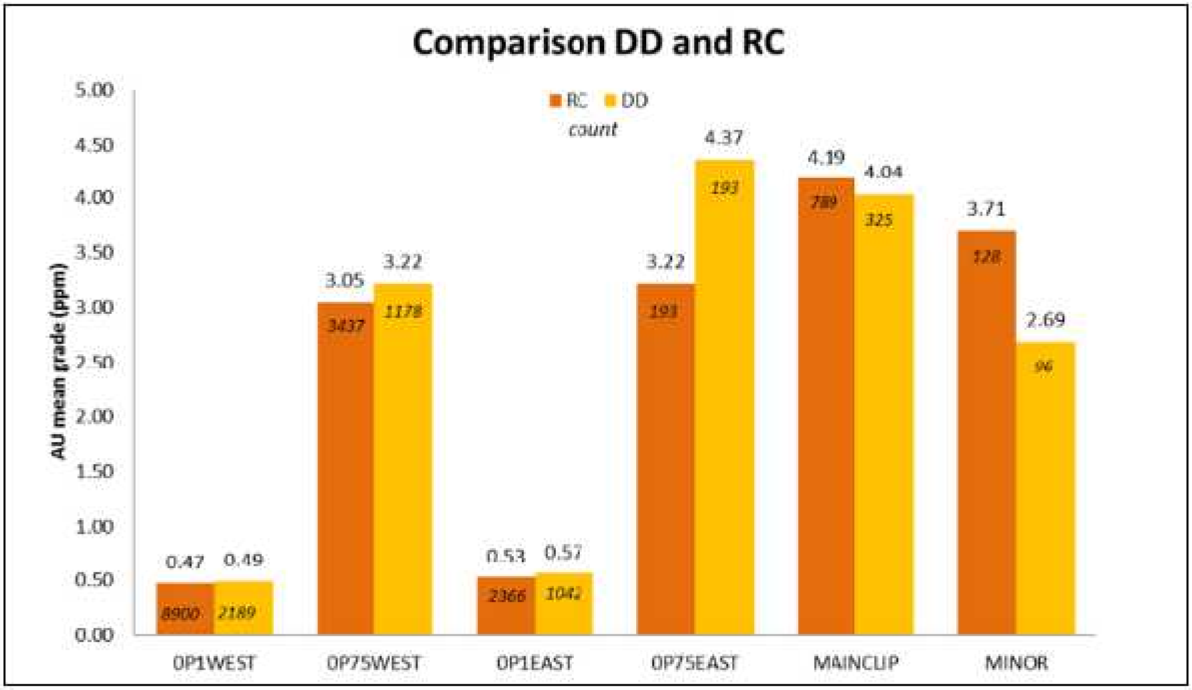


Figure 14-18: Comparison of unweighted Au mean grade per drilling type

The samples from DD, RC and RCGC were used for the estimation.

14.13 Summary Statistics

This section compares the mean gold grade, declustered grade and top-cut grade within each domain (Table 14-11).

Table 14-11: Comparison of mean gold grade (g/t) within each domain

Domains	AU mean grade (g/t)		
	Unweighted	Weighted	Top cut applied
0P1WEST	0.54	0.47	0.47
0P75WEST	4.61	3.11	3.11
0P1EAST	0.64	0.55	0.54
0P75EAST	3.71	3.77	3.77
MAIN	6.50	5.67	4.13
MINOR	5.72	3.26	3.20

14.14 Variography

Spatial variability was studied for the two main elements gold and arsenic within each domain. The variography of gold and arsenic at Maud Creek was completed using the Isatis software, produced by Geovariance.

The choice of variogram directions were constrained by a combination of drilling data, geological continuity and domains. The structures observed were poor in most domains. To improve the variography study, the Gold grade was transformed into a Gaussian value within each domain while the arsenic was only transformed into a Gaussian value in domain 0P1EAST and 0P1WEST. The transformation was completed using a punctual Gaussian anamorphosis. The number of Hermite polynomial was adjusted according to the domain and element, varying between 30 and 70.

The variogram was computed on the 1 m Gaussian composites within each domain separately.

A model was fitted to the Gaussian variogram and back transformed to a model representing the composite data.

The parameters for the variogram model of the gold and arsenic for all the domains are given in Table 14-12 and Table 14-13. As expected, the Au element is fairly variable with a nugget that accounts for at least 44% of the total sill.

In MAIN domain, the best model is reasonably well described by two spherical structures and a nugget for both elements. The nugget for the Au variogram account for 74% of the total sill while the nugget for the arsenic variogram account for only 4%. The first structure is relatively short. Figure 14-19 shows the variogram fitting within the MAIN domain for gold and arsenic.

Table 14-12: Variogram model characteristics for Au

Domains	Rotation Geologist Plane			Structures	Sills	Range		
	AZI	DIP	PITCH			U	V	W
0P1WEST	0	67	161	Nugget	0.97			
				Spherical 1	0.44	30	30	5.6
0P75WEST	0	67	90	Nugget	16.42			
				Spherical 1	4.97	35	35	2.8
				Spherical 2	3.92	35	35	12
				Spherical 3	2.31	90	35	12
0P1EAST	0	65	95	Nugget	2.7			
				Spherical 1	0.62	20	20	7
0P75EAST	Omni-directional			Nugget	14.23			
				Spherical 1	3.48	110	110	-
MAIN	0	65	116	Nugget	625.12			
				Spherical 1	162.15	40	40	20
				Spherical 2	60.73	160	100	20
MINOR	0	65	116	Nugget	13.35			
				Spherical 1	9.46	40	40	20
				Spherical 2	7.79	160	100	20

Table 14-13: Variogram model characteristics for Arsenic

Domains	Rotation (degree) Geologist Plane			Structures	Sills (g/t)	Range (m)		
	AZI	DIP	PITCH			U	V	W
0P1WEST	0	100	-180	Nugget	266,079			
				Spherical 1	456,877	40	20	10
				Spherical 2	402,390	200	80	50
0P75WEST	0	67	180	Nugget	1,000,000			
				Spherical 1	5,200,000	25	40	9
				Spherical 2	2,000,000	140	40	40
0P1EAST	0	70	-180	Nugget	1,840,481			
				Spherical 1	2,526,700	50	40	35
				Spherical 2	2,001,163	100	40	35

0P75EAST	Omni-directional			Spherical 1	11,050,000	19	19	-
MAIN	0	85	170	Nugget	300,000			
				Spherical 1	5,500,000	35	35	15
				Spherical 2	2,000,000	100	80	15
MINOR	Omni-directional			Nugget	800,000			
				Spherical 1	4,000,000	40	40	40

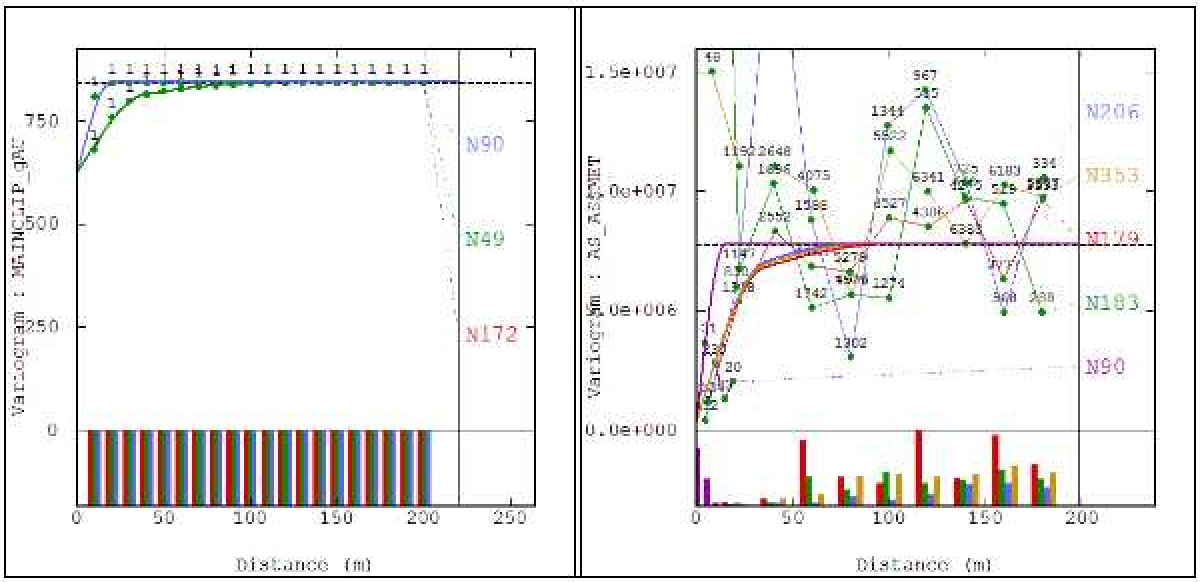


Figure 14-19: MAIN variogram fitting – left: Au, Right: As

14.15 Estimation

14.15.1Gold and Arsenic

The Ordinary Kriging (OK) method was selected to estimate gold and arsenic within each domain. Domain boundaries are considered hard, with two exceptions.

Composites from the MAIN were allowed to be used in the estimation of the MINOR domain (soft boundary, although the domains are not in direct contact).

To address the potential under call in ounces where the 0.75 g/t Leapfrog grade shells did not connect, a two stage process was used to estimate the 0.1 g/t Au domain. Firstly the 0.1 g/t Au domain was estimated using a hard boundary constrained between 0.1 and 0.75 g/t Au material. Secondly, and overriding the first estimation, a soft boundary was estimated using all of the material located outside the main vein (primary). A tight search ellipse orientated perpendicular to the dip direction was employed, to ensure only those blocks within the 0.1 g/t Au domain, directly adjacent to the main vein, were informed during the second estimation. (The overall effect of this methodology was to increase the 0.1 g/t Au domain from an average grade of 0.48 g/t to 0.7 g/t Au, with some extra 140 koz at zero cut of.)

Identified outliers were replaced by the top cut when they exceeded a threshold distance of 15 m from the block centroid. Within 15m the full uncut value was used.

The estimation was done using one neighbourhood search size specific to each domain. The maximum number of samples selected for the estimation is a compromise between the best local smoothed estimates, using many samples to optimise the slope of regression minimise negative weights and the best global grade and tonnage curve, de-smoothed, estimate using fewer samples. Details of the search ellipsoids for gold are shown in Table 14-14 and Table 14-15. Details of the search ellipsoids for arsenic are shown in Table 14-16. Note that the maximum number of samples is given by the number of sectors multiplied by the optimum number of sample per sector.

Table 14-14: Neighbourhood parameters – Au estimated by OK

Domains	0P1WEST	0P75WEST	0P1EAST	0P75EAST	MAIN	MINOR
Rotation						
X Angle	0	0	0	0	0	0
Y Angle	67	67	65	65	67	67
Z Angle	161	90	95	95	116	116
Search Ellipsoid						
X max	400	400	400	400	600	600
Y max	400	400	400	400	600	600
Z max	200	200	200	200	300	300
Number of sectors	8	8	10	8	1	1
Minimum samples	8	8	8	8	8	8
Optimum samples per sector	4	4	3	4	32	32
Minimum distance between data				0.7		
Cut-off						
Threshold	50		40		200	50
Distance	15		15		15	15

Table 14-15: Neighbourhood parameters – Au estimated by OK – 0P1WEST soft boundary

Domains	0P1WEST (Soft boundary)
Rotation	
X Angle	0
Y Angle	67
Z Angle	161
Search Ellipsoid	
X max	400
Y max	400
Z max	10
Number of sectors	8
Minimum samples	4
Optimum samples	4
Minimum Number samples per line	2
Cut-off	
Threshold	50
Distance	15

Table 14-16: Neighbourhood parameters – AS estimated by OK

Domains	0P1WEST	0P75WEST	0P1EAST	0P75EAST	MAIN	MINOR
Rotation						
X Angle	0	0	0	0	0	0
Y Angle	100	67	70	65	85	67
Z Angle	-180	90	-180	95	170	116
Search Ellipsoid						
X max	400	400	400	400	600	600
Y max	400	400	400	400	600	600
Z max	200	200	200	200	300	300
Number of sectors	8	4	4	8	4	1
Minimum samples	40	1	1	40	1	40
Optimum samples per sector	10	8	20	10	10	400
Minimum distance between data				0.7		
Maximum Distance without sample	50				90	
Cut-off						
Threshold				12000		10000
Distance				15		15

14.15.2 Other elements

The Inverse Distance of order 2 (ID2) method was selected to estimate the other elements in the western domains and MAIN while the Nearest Neighbourhood (NN) method was selected to estimate in the eastern domains and MINOR. ID2 and NN characteristics are as follow:

- Domain boundaries are considered hard, at the exception of the contact between MINOR and MAIN when estimating in the MINOR domain and the eastern domains.
- Identified outliers were replaced by the top cut if outside the threshold distance.
- The search orientation and size were similar for all domains. The number of data used for the estimation is limited by the neighbourhood parameters. Details of the search ellipsoids are shown in Table 14-17.
- Due to the limited number of data, a restriction on the distance without samples was applied meaning that not all the block cells were informed during the ID2 or NN estimation.

Table 14-17: Neighbourhood parameters

Domains	0P1WEST	0P75WEST	0P1EAST	0P75EAST	MAIN	MINOR
Rotation						
X Angle	0					
Y Angle	90					
Z Angle	180					
Search Ellipsoid						
X max	400					
Y max	400					
Z max	200					
Number of sectors	4					
Minimum samples	10					
Optimum samples	10					
Maximum Distance without sample	60	60	30	30	60	50

14.16 Density

There are three periods of density sampling over the life of the project as recorded in the density database supplied by Newmarket Gold. Around 1991 Places made 43 measurements from the WD series holes from 0.1 to 0.3m lengths of core. Around 1995 Kalmet took 481 measurements from the MD series holes from 0.1m to 0.3m intervals of core. In 2011 Newmarket Gold took 2.145 measurements from 0.1m lengths of core from the MC series holes. The Newmarket Gold measurements used some half core and some full core.

There are a total of 2.669 density measurements available, 740 of which are in mineralization (as defined by the estimation domains). Oxide states show significant differences in density (Table 14-18 and Table 14-19). There is no practical difference in densities within the fresh material for lithology or estimation domain (Table 14-20, Table 14-21 and Table 14-22). The densities shown in Table 14-18 have been used to inform both the resource and waste model.

Table 14-18: Density statistics by oxide state all data

Oxide state	Count	Density
1 (completely oxidised)	3	2.11
2 (oxide / transition)	35	2.60
3 (transition)	53	2.72
4 (transition/ fresh)	39	2.77
5 (fresh)	2538	2.80

Table 14-19: Density statistics by oxide state in all estimation domains

Oxide state	Count	Density
1 (completely oxidised)	2	2.10
2 (oxide / transition)	23	2.57
3 (transition)	16	2.65
4 (transition/ fresh)	3	2.77
5 (fresh)	696	2.79

Table 14-20: Density statistics by estimation domain within all estimation domains in Fresh rock

Domain	Count	Density
0.1 halo	249	2.80
0.75 halo	327	2.80
Veins	120	2.78

Table 14-21: Density statistics by lithology in Fresh rock all data

Lithology	Count	Density
Tuff	1016	2.79
Sediments	640	2.59
Minor Vein	30	2.76
Major Vein	90	2.78
Dolerite	762	2.85

Table 14-22: Density statistics by lithology in estimation domains in in Fresh rock

Lithology	Count	Density
Tuff	387	2.81
Sediments	160	2.77
Minor Vein	31	2.76
Major Vein	90	2.78
Dolerite	29	2.82

14.17 Validation

Following the estimation, the final model was reviewed and validated. The estimation validation consists of statistical comparison, visual comparison and swath plots. The estimation validation of the OK includes also the review of the kriging output variables such as the regression slopes.

The regression slopes of the Au estimates were relatively poor with a high number of values lower than 0.5. Figure 14-20 shows the results for the gold and arsenic within the MAIN domain.

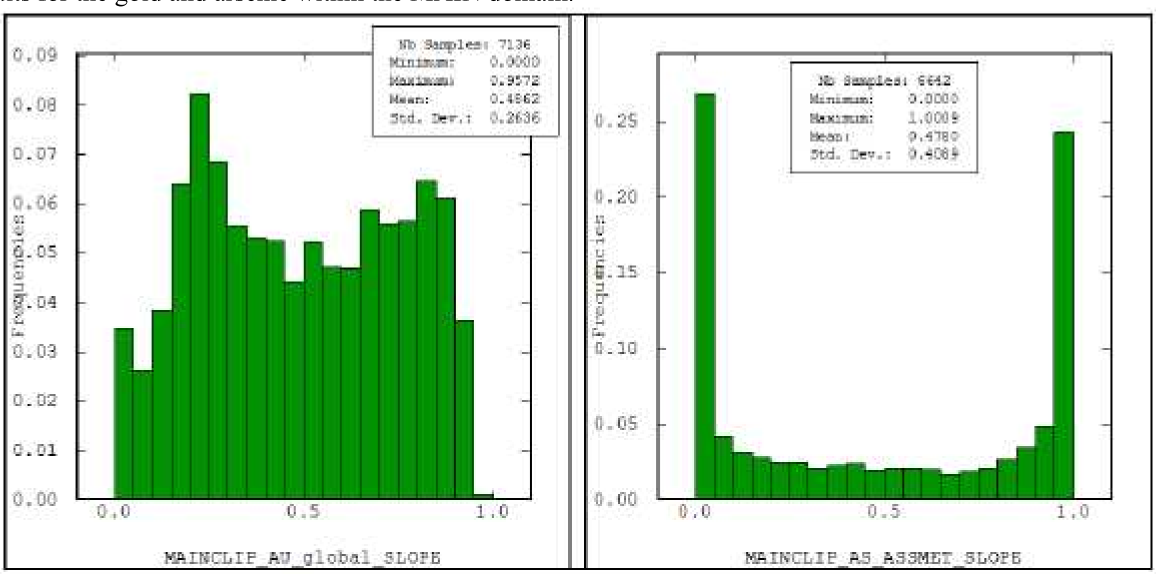


Figure 14-20: Regression slope histogram for Au estimates (left) and AS estimates (right)

The gold estimation by OK was re-run by increasing the maximum allowable number of data used in the search neighbourhood to 400. Increasing the maximum number of samples used for estimation in the neighbourhood definition improves the local block OK performance and results in lower slopes of regression overall (Figure 14-21) but over smooths the block distribution such the grades and tonnages at higher cut-offs are not realistic (Figure 14-23). For identification purposes in this section of the report, the check run OK is referred to Local OK while the gold estimation is called Global OK. Table 14-23 compares the global statistics between the composite top cut data and the two gold estimates. Global statistics showed reasonable good correlation at zero cut-off between the estimates and the data, slightly better for the Global OK estimates.

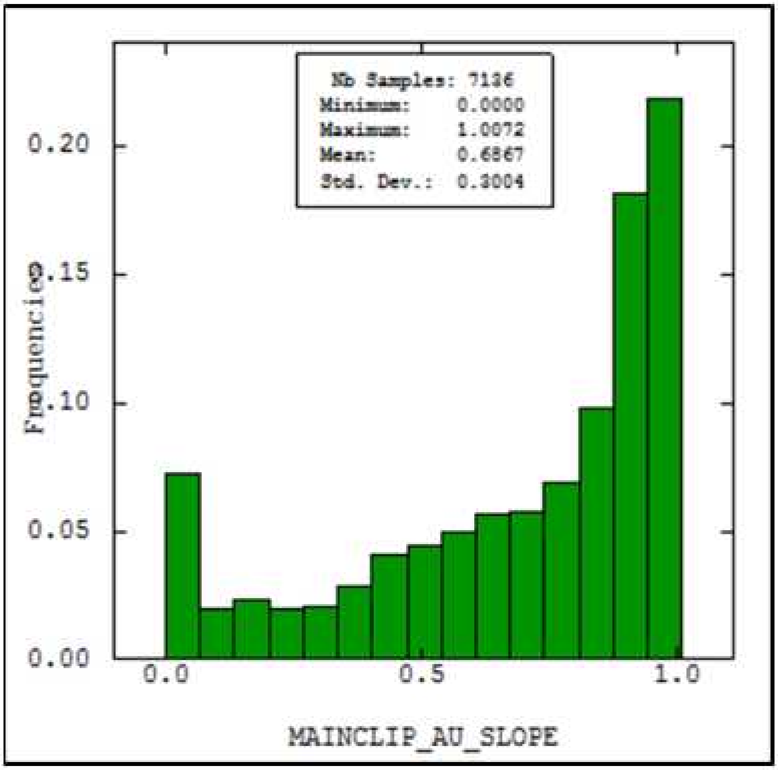


Figure 14-21: Regression slope histogram for Main Au Local check estimate

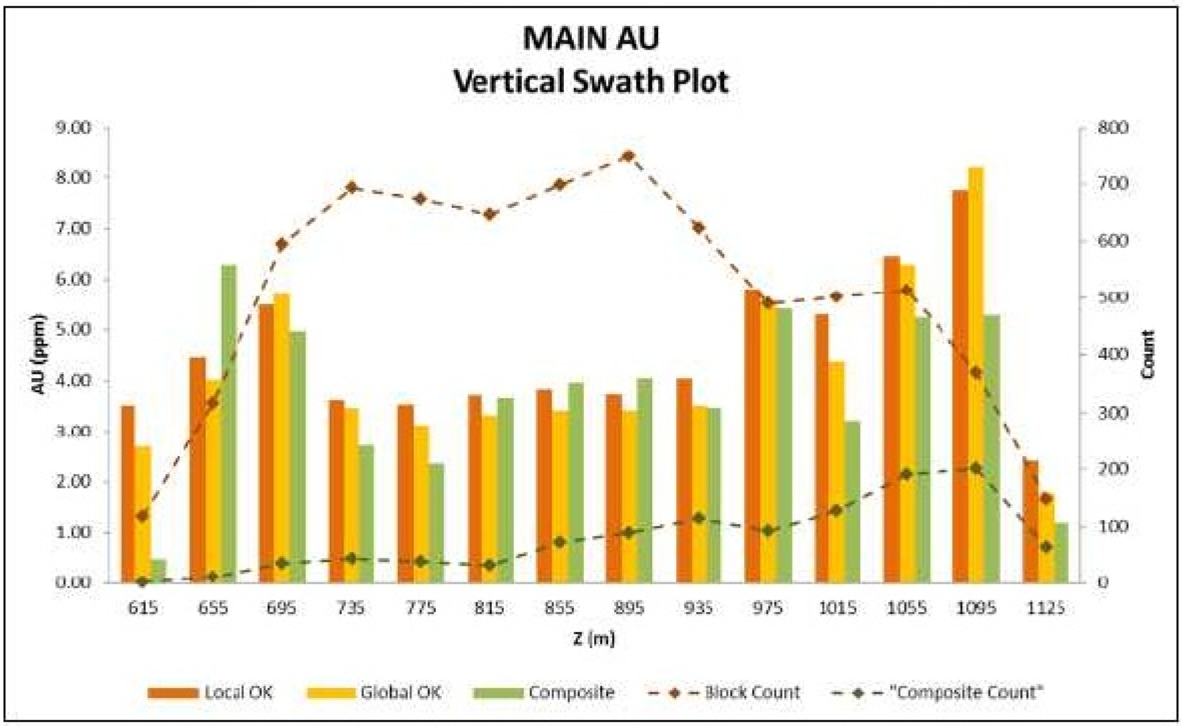
Table 14-23: Comparison of declustered top cut Au composite and estimates per domain

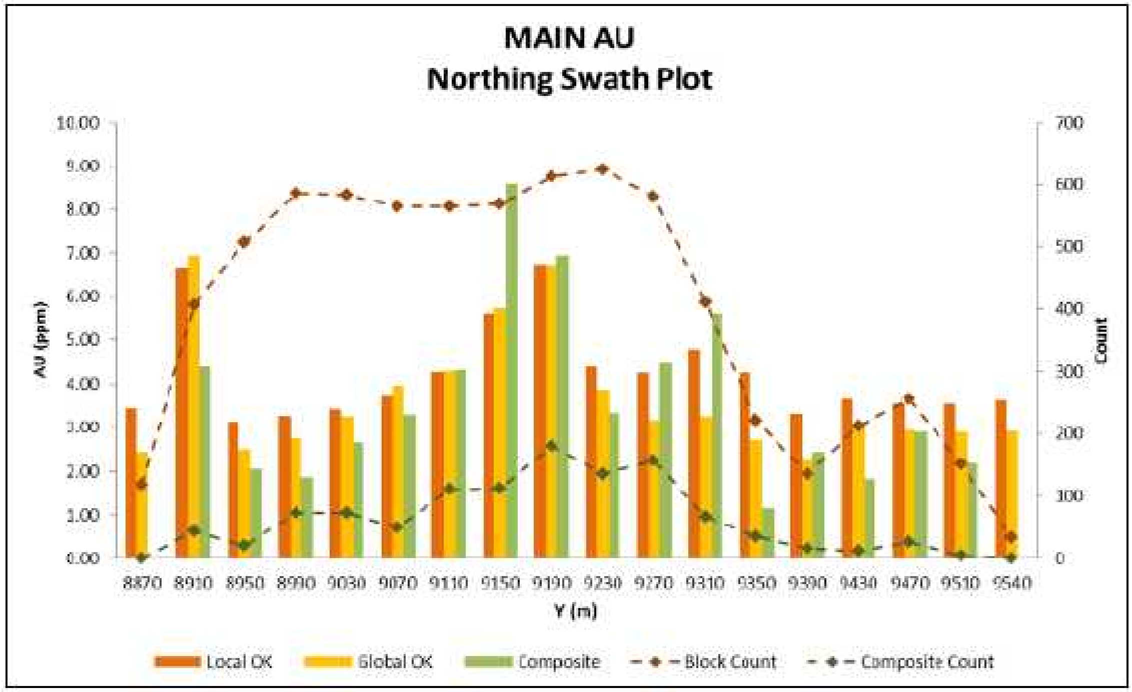
Domains	Variable	# Samples/ blocks	Minimum	Maximum (Cut max)	(Declustered) Mean	% Difference
0P1WEST	Composite	11,091	0.001	58.18 (50)	0.47	
	Local OK	28,831	0.13	7.13	0.48	2.1
	Global OK	28,831	0.11	7.14	0.46	-2.1
0P75WEST	Composite	4,615	0.05	282.83	3.11	
	Local OK	8,650	0.99	25.70	3.33	7.1
	Global OK	8,650	0.95	28.61	3.34	7.4
0P1EAST	Composite	3,408	0.01	76.07 (40	0.54	
	Local OK	12,592	0.18	6.03	0.58	7.4
	Global OK	12,592	0.11	7.55	0.54	0.0
0P75EAST	Composite	386	0.01	25.38	3.77	
	Local OK	992	1.71	10.91	4.06	7.7
	Global OK	992	1.29	11.13	3.98	5.6

Domains	Variable	# Samples/ blocks	Minimum	Maximum (Cut max)	(Declustere d) Mean	% Difference
MAIN	**Composite**	1,114	0.005	619.43 (50)	4.13	
	Local OK	7136	0.63	91.87	4.32	4.6
	Global OK	7,136	0.06	103.36	4.30	4.1
MINOR	**Composite**	224	0.01	79.67 (50)	3.20	
	Local OK	1,880	0.228	24.874	3.65	14.1
	Global OK	1,880	1.499	22.599	3.96	23.8

At 0.0 g/t cut-off grade

Within the MAIN domain, 40 m swath plot in X, Y and Z directions showed a reasonably good correlation within area well sampled (Figure 14-22). In areas with many composite samples, the OK has smoothed the grade while in area with limited composite samples the OK tends to overestimate. The Local OK is mostly over-estimating the grade compared to the Global OK approach.





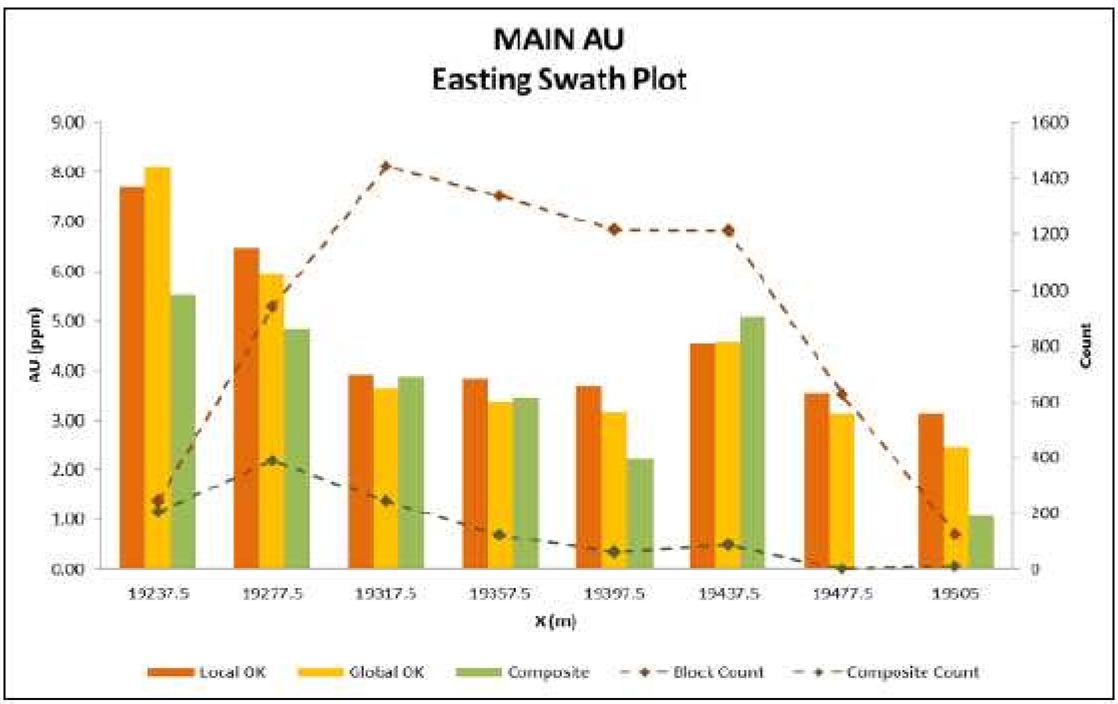


Figure 14-22: MAIN - Au Global OK - Swath plots

The gold estimations were then compared against the theoretical block distribution defined by a change of support calculation on the Gaussian anamorphosis of the sample distribution. Figure 14-23 compares the estimates with the theoretical block distribution for gold in the MAIN domain. The Local OK displays a sharp tonnage and grade gradient between gold grade cut-off of 3g/t and 4g/t. The grade tonnage curve for Global OK shows a better correlation with the theoretical block distribution. The metal tends to be consistently under-estimated compared to the theoretical block distribution. The theoretical block distribution is considered to predict tonnage, grade and metal closer to the expected global recovery. Therefore the Global OK is considered a better estimate at this stage of the mining project. All further references to the gold estimate in this report refer to the Global OK Au estimate.

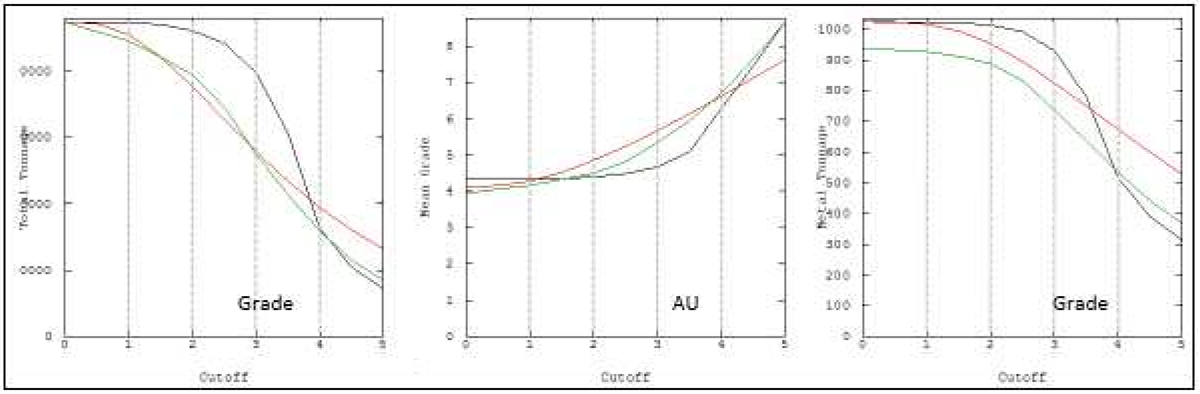


Figure 14-23: MAIN - gold - Grade tonnage comparison Reasonable Prospects of economic extraction

From Left to right: Tonnage, Grade, Metal, Green = Global OK, Black= Local OK and Red= Theoretical Block distribution

In assessing the criteria for reasonable prospects of economic extraction both open pit and underground scenarios were considered. With respect the scattered lower grade mineralization contained within the eastern domains (0p1E and 0p75E) near surface a simple pit optimisation using the optimistic parameters in Table 14-24 at twice the current gold spot price did not generate a pit of practical size on the eastern domains. All material in the eastern domains is not considered to have reasonable prospects of economic extraction and does not appear in the Mineral Resource.

Table 14-24: Pit optimisation parameters for evaluation of Mineral Resource classification

Parameter	Value
Gold Price (twice current spot)	AUD2,830 /Oz
Processing Cost (Whittle)	AUD58.4 /t mill feed
Recovery Oxide and Transition	81%
Recovery Fresh	90%
Overall wall slope angles	34 degrees

With respect to the underground potential the grade is reasonably consistent down to approximately 650 mRL below which it drops significantly. All material below 650 mRL is not considered to have reasonable prospects of economic extraction and does not appear in the Resource.

14.18 Classification

Classification is based on a combination of drill spacing, geological interpretation confidence, proximity to the previously mined open pit, reasonable prospects of economic extraction and grade. The classification areas are coherent zones and do not contain isolated blocks of lower classifications within them.

Measured is defined by the main, minor and 0p75W domains above 950 mRL and between plunging north and south boundaries that approximate the limits of the closer drill spacing (Figure 14-24).

Indicated (Figure 14-25) is defined by:

1. The 0p1 domain external to the Measured above 950 mRL; and

2. The approximate limits of the 20 x 20 m drilling.

The inferred is the remaining material above 650 mRL within approximately 50 m of drilling and the areas with low geological confidence in the orientations of the controls on mineralization and assumed structures (Figure 14-26).

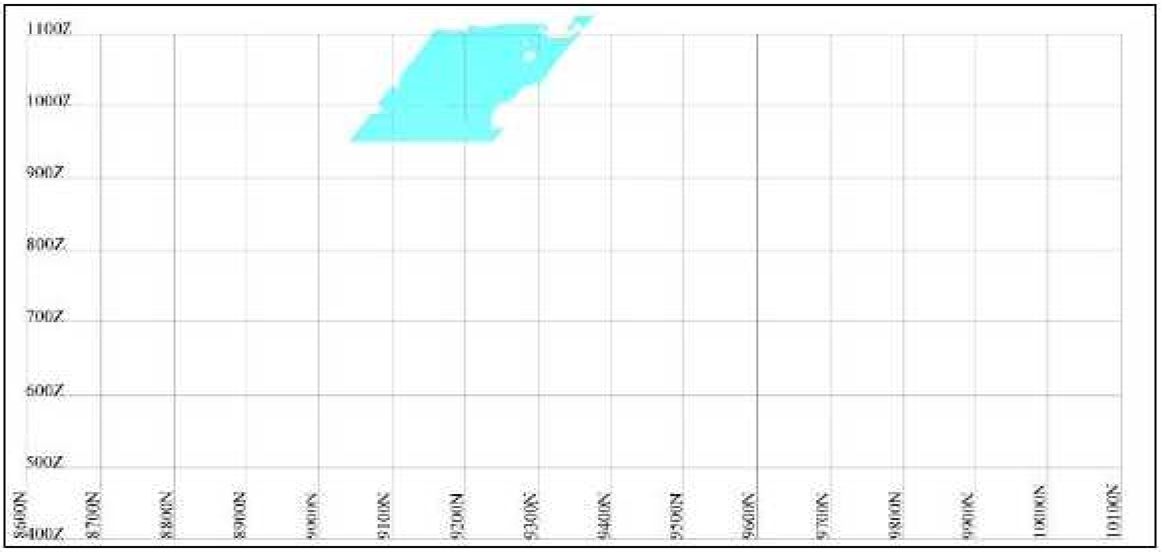


Figure 14-24: Long section facing west displaying measured blocks for all domains

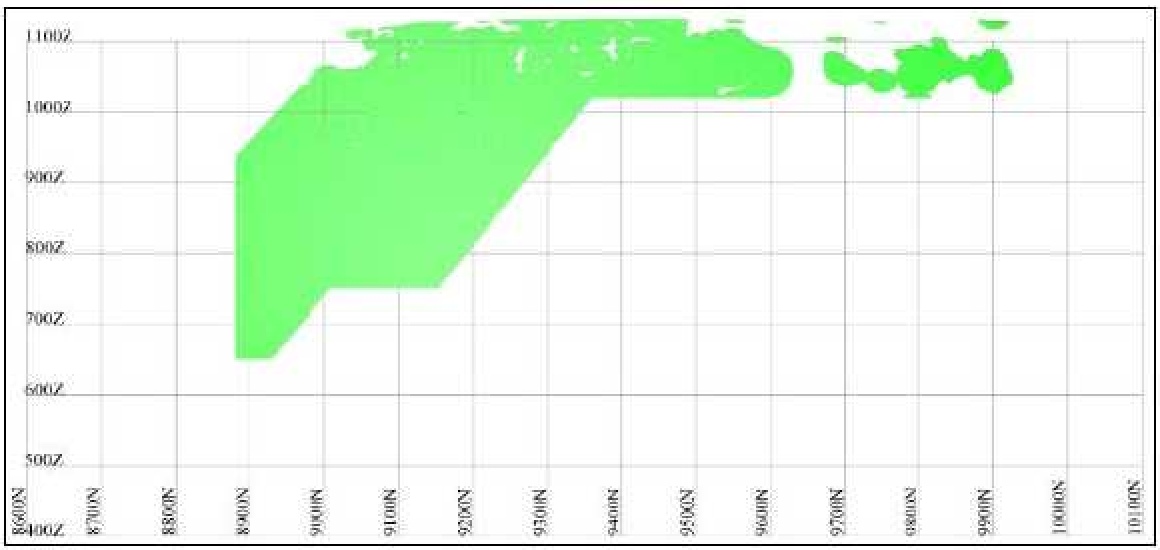


Figure 14-25: Long section facing west displaying indicated blocks for all domains

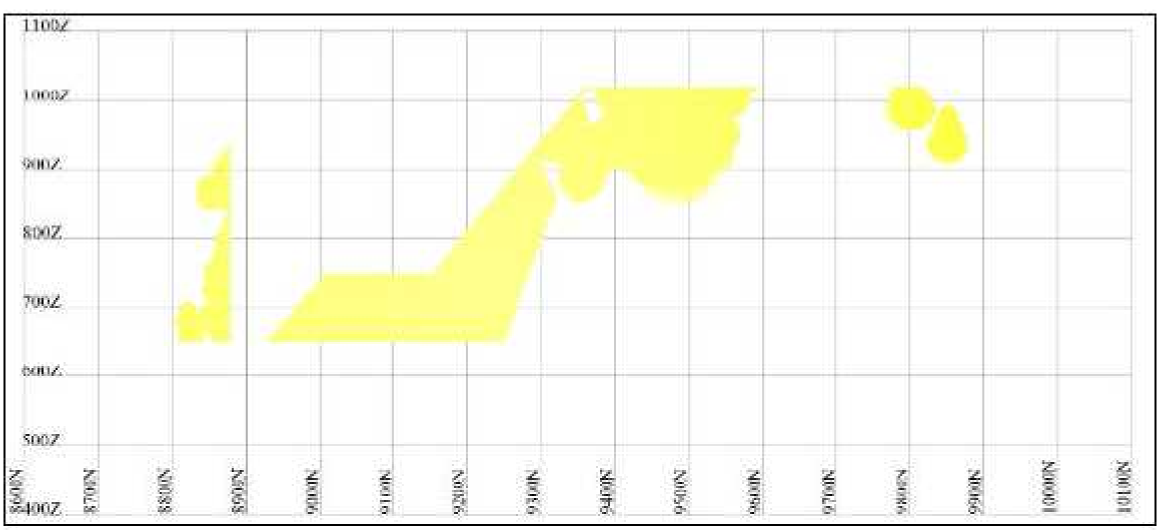


Figure 14-26: Long section facing west displaying inferred blocks for all domains

14.19 Mineral Resource Tonnage and Grade

Given the Mineral Resource is amenable to open cut and underground mining a split set of cut-offs is used for reporting. An elevation limit of 950 mRL has been used for the depth limit for open cut reporting as this is 50 m below the simplistic whittle pit optimisation depths generated with optimistic revenues.

A cut-off of 0.5 g/t Au is defined as the base case (Table 14-25), a comparison at 1.0 g/t Au is also included (Table 14-26). The open pit Mineral Resource is exclusive of the underground Mineral Resource. The Mineral Resources are stated here for the Maud Creek deposit with an effective date of 15 March 2016

Table 14-25: Open pit Mineral Resource above 950 mRL at 0.5 g/t Au cut-off – base case

Classification	Tonnage (kt)	Grade (Au g/t)	Contained Metal (KOz Au)
Measured	1,070	5.6	190
Indicated	1,100	2.1	75
Measured and Indicated	**2,170**	**3.84**	**268**
Inferred	530	1.4	25

It should be pointed out the Mineral Resource estimate is categorized as Measured, Indicated and Inferred as defined by the CIM guidelines for resource reporting. Mineral resources do not demonstrate economic viability, and there is no certainty that these Mineral Resources will be converted into mineable reserves once economic considerations are applied. The Measured, Indicated and Inferred Mineral Resource estimate has been prepared in compliance with the standards of NI 43-101 by Danny Kentwell, FAusIMM.

Notes to Table 14-15:

1. CIM definitions followed for classification of Measured, Indicated, and Inferred Mineral Resources.
2. Mineral Resources estimated as of 15 March 2016.
3. Mineral Resources stated according to CIM guidelines.
4. Totals may appear different from the sum of their components due to rounding.
5. Reported at a 0.5 g/t cut-off grade.
6. The open pit Mineral Resource is exclusive of the underground Mineral Resource.
7. The Mineral Resource estimation was performed by Danny Kentwell FAusIMM fulltime employee of SRK Consulting, who is a Qualified Person under NI 43-101.

Table 14-26: Open Pit Mineral Resource above 950 mRL at 1.0 g/t Au cut-off – comparison only

Classification	Tonnage (kt)	Grade (Au g/t)	Contained Metal (koz Au)
Measured	1 067	5.59	192
Indicated	1 100	2.14	76
Measured and Indicated	**2167**	**3.84**	**268**
Inferred	232	2.36	18

The underground Mineral Resource consists only of material below 950 mRL. The base case is stated at 1.5 g/t Au cut-off (Table 14-27). A comparison at 2.0 g/t Au cut-off is provided in Table 14-28. The underground Mineral Resource is exclusive of the open pit Mineral Resource.

Table 14-27: Underground Mineral Resource below 950 mRL at 1.5 g/t Au cut-off - base case

Mineral Resource Classification	Tonnage (kt)	Grade (Au g/t)	Contained Metal (koz Au)
Measured	-	-	-
Indicated	4,330	3.28	456
Measured and Indicated	**4,330**	**3.28**	**456**
Inferred	1,450	2.65	124

It should be pointed out the Mineral Resource estimate is categorized as Measured, Indicated and Inferred as defined by the CIM guidelines for resource reporting. Mineral resources do not demonstrate economic viability, and there is no certainty that these Mineral Resources will be converted into mineable reserves once economic considerations are applied. The Measured, Indicated and Inferred Mineral Resource estimate has been prepared in compliance with the standards of NI 43-101 by Danny Kentwell, FAusIMM.

Notes to Table 14-27 and 14-28:

1. CIM definitions followed for classification of Measured, Indicated, and Inferred Mineral Resources.
2. Mineral Resources estimated as of 15 March 2016.
3. Mineral Resources stated according to CIM guidelines.
4. Totals may appear different from the sum of their components due to rounding.
5. Reported at a 1.5 g/t cut- off grade.
6. The underground Mineral Resource is exclusive of the open pit Mineral Resource.
7. The Mineral Resource estimation was performed by Danny Kentwell FAusIMM fulltime employee of SRK Consulting, who is a Qualified Person under NI 43- 101.

Table 14-28: Underground Mineral Resource below 950 mRL at 2.0 g/t Au cut-off - comparison only

Mineral Resource Classification	Tonnage (kt)	Grade (Au g/t)	Contained Metal (KOz Au)
Measured	-	-	-
Indicated	3,490	3.65	410
Measured and Indicated	**3,490**	**3.65**	**410**
Inferred	1,026	3.04	100

It should be pointed out the Mineral Resource estimate is categorized as Measured, Indicated and Inferred as defined by the CIM guidelines for resource reporting. Mineral resources do not demonstrate economic viability, and there is no certainty that these Mineral Resources will be converted into mineable reserves once economic considerations are applied. The Measured, Indicated and Inferred Mineral Resource estimate has been prepared in compliance with the standards of NI 43-101 by Danny Kentwell, FAusIMM.

15 Mineral Reserve Estimate

Mineral Reserves are not being reported.

16 Mining Methods

The open pit design work carried out by SRK was based on the preferential allocation of potential mill feed to the proposed underground operations and oxide mineralization being treated as waste. With the proposed treatment of mineralization at the Union Reefs Processing Plant, the Oxide or contained within the pit design was report and included in the proposed mill feed.

Considering oxide as income generating provides the potential to increase the size of the open pit. The impact of the project economics are potentially large, particularly if the constraints of the waste dumps or underground preferences are relaxed.

Should the project proceed SRK recommends that a range of sensitivities/ scenarios are considered at a high level to inform the project team of the implications of these constraints. The sensitivities should include combinations of considering the open pit oxide mineralization as a mill feed source with both the underground preferences retained and removed. Conventional open pit mining techniques are proposed.

The underground is designed using conventional bottom-up open stoping mining methods utilising a combination of cemented rock fill and waste fill. The stopes will be accessed via a decline and level intervals and stoping lengths based on the geotechnical guidelines discussed below.

At this stage of the study, the design includes decline access from a portal located on surface outside the footprint of the open pit. The opportunity exists to modify the design to interact with the open pit design but this will potentially have constraints on production continuity.

16.1 Geotechnical

In conjunction with the Mineral Resource review, a comprehensive review was undertaken of the available data and analysis to determine the open pit and underground geotechnical design guidelines.

An assessment of overall slope angles and underground mining parameters has been undertaken by SRK using geological and geotechnical drilling data supplied by Newmarket Gold. The analysis provides good early-stage design guidelines of the geotechnical properties of the rock mass. The typical geotechnical conditions on site can be summarised as follows:

- The Hanging wall Tuffs are typically massive but may be locally bedded. Hanging wall tuffs are also affected by the numerous shears present in the Hanging wall, resulting in reduced strength, increased fracture frequency and graphitic and/ or chloritic alteration of the rock mass.
- The Footwall Sediments consist of low to medium strength thinly bedded or laminated mudstone and siltstone, and medium to thickly bedded sandstone. Zones of intense shearing with chlorite and graphite alteration occurring in the 5 to 10 m below the mineralized zone where the sediments are commonly black, highly graphitic and/or chloritic, very weak and fissile.
- The competency of the mineralized zone can be expected to be variable with competent, partially silicified mineralized zones separated by zones of intensely sheared rock.
- The distribution of the various fault configurations is not understood at this stage and this should be one of the main focus for subsequent field investigations.

16.1.1 Level of Confidence

The perceived level of confidence in the different data streams available were rated subjectively, as shown in Table 16-1. A five-point rating scale of Very Low – Low – Moderate – High – Very High has been used.

At this stage of the project development, at least a Low confidence rating should be expected for all items, with all items requiring further investigation. Aspects for which no data is currently available or represent a key concern have been flagged as Very Low.

Table 16-1: Qualitative risk assessment of study components

Data	Confidence level
Empirical: rock mass characterisation	Low
Structural: major structures	Low to Moderate
Structural; rock mass structures	Low
Rock mass strengths	Very low
Rock material strengths	Very low
Groundwater conditions	Very low
Slope angle recommendations	Low

16.1.2 Project Risks and Opportunities

The main project risks relate to the absence of data required for the development of design parameters. The areas where data is required are:

Minor geological structures – additional pit mapping/photogrammetry and diamond drilling with geotechnical and structural logging are necessary to overcome the current orientation bias and increase the data density throughout the study area.

Rock mass characterisation and strength – further drilling and evaluation is needed to improve confidence in the rock mass properties of all the domains, in particular within the major fault zones.

Rock material strength – a representative laboratory testing program needs to form part of the next stage of investigation. This should allow the identification and evaluation of design mechanical properties which can be used for numerical analysis of pit designs at the next stage of the study.

Improved understanding of the project groundwater conditions is needed to understand the interaction between the site hydrogeology and the pit walls.

16.1.3 Review of Geotechnical data

Geotechnical data was collected from drillholes drilled between 1990 and 2011. The geotechnical data was collected from the deposit and the rock mass adjacent to the deposit, with the more recent drilling collecting data throughout the wider area. Recent drilling was also more likely to include a wider range of geotechnical parameters. Sediments located in the footwall of the deposit are not well represented within the overall geotechnical dataset. Additional geotechnical drillhole data will be required for future studies. Geotechnical laboratory testing will be required from samples recovered from future drilling.

16.1.4 Rock mass characterisation

Geotechnical domains have been determined for both the open pit and underground areas at the Maud Creek project based on geological units, weathering, structural setting and rock mass quality. Figure 16-1 shows the different geotechnical domains selected for the open pit analysis. For underground component of the study, stope stability assessments considered domains within the deposit and 10 m either side of the deposit only, whilst broader domains were used to characterise the rock mass for ground support requirements.

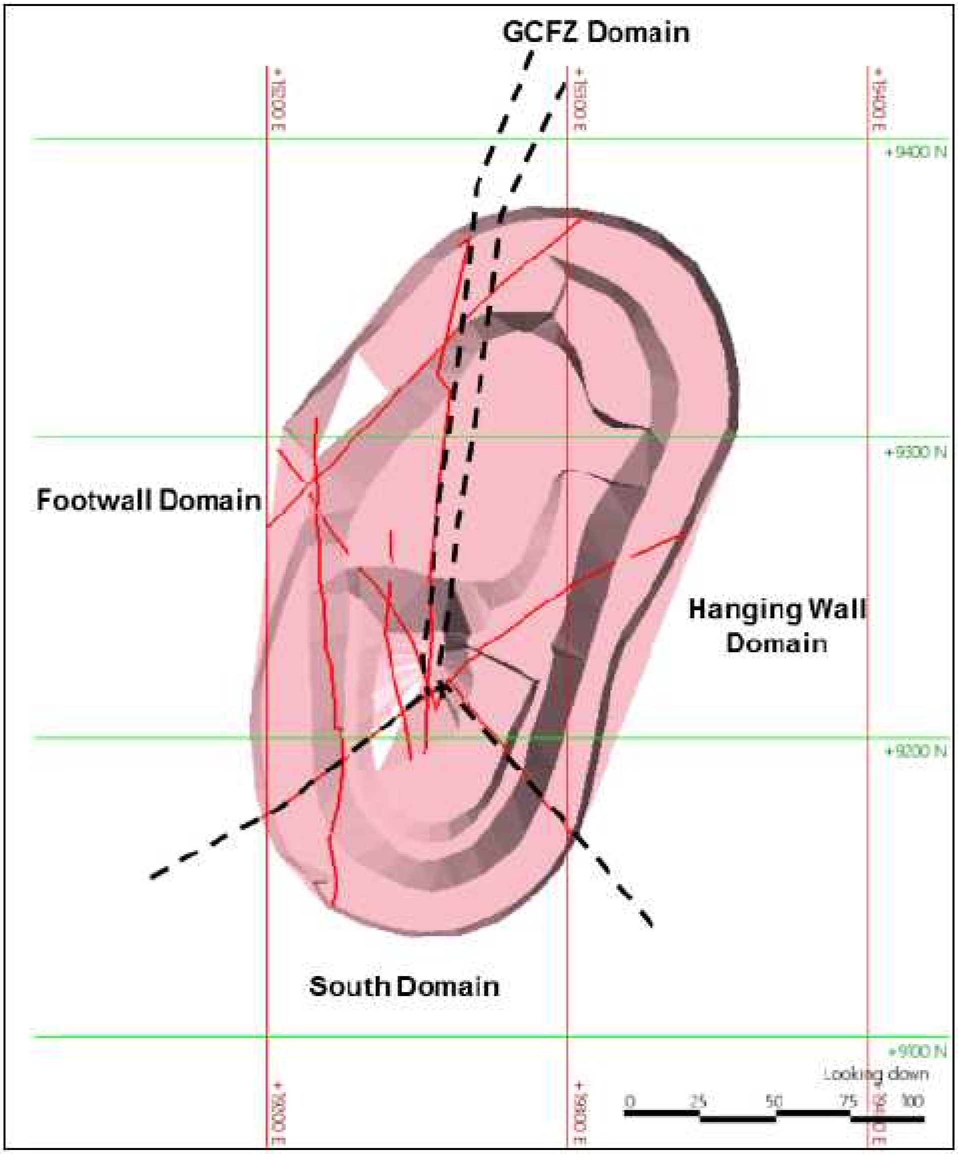


Figure 16-1: Plan view of existing pit showing the geotechnical domains used for rock mass characterisation

16.2 Underground Design Considerations

Newmarket Gold requested that SRK Consulting (Australasia) Pty Ltd (SRK) investigate the potential for processing mineralization from the Maud Creek deposit at the existing Union Reefs facility to eliminate the capital cost of constructing a process plant at Maud Creek. Offsite processing would likely result in tailings becoming unavailable and therefore paste fill no longer being a viable backfilling material. Modifications to the backfilling strategy, mining method and sequence will form part of the investigation.

16.2.1 Mining Method

The proposed mining direction is a bottom up sequence retreating to a central access and utilising waste rock fill with cemented rock fill in strategic locations. The mining direction requires the separation of the deposit into multiple mining areas to provide earlier mill feed.

The establishment of three mining areas has been considered and would require the creation of two artificial sill pillars at the base of each mining area to allow the later extraction of the pillar separating the areas. The artificial sill pillars are proposed to be constructed by placing cemented rock fill (CRF) as backfill in the stopes on the lowest level in each mining area. It is recommended that the bottom level of the mine be filled with CRF to allow mining at depth at a later stage.

16.2.2 Stope Stability

The change of mining method and modification of mining sequence will not change the stable stope dimensions. However the changes may require the final stopes in each level have longer strike lengths for a more practical final extraction. The longer strike lengths would be approximately 20-25m. The longer strike lengths would affect stope stability in footwall sediments, and also exceed the strike length design recommendations of 10m and 15m for footwall sediment domains A and B respectively (Table 16-2) (SRK 2015).

Stope stability and overbreak for the larger central stopes can be managed by designing stable beams between the stope footwall and shear zones or by installing cablebolts in the footwall or hanging wall. The equivalent linear overbreak slough (ELOS) for the increased strike lengths would be up to 2.0m for an unsupported stope in the footwall sediments domains A and B. ELOS for other geotechnical domains remains unchanged.

Stope dilution may be controlled by designing stable beams between the stope footwall and the shear zones, reduction in stope strike spans or cable bolt reinforcement of stope hanging wall or footwall.

Table 16-2: Stope Geometry

Domain	N'	Hydraulic Radius (HR)		Strike length (m)	ELOS (m)	ELOS 25 m strike (m)	Stope Height (m)
		Unsupported	Supported				
Sediments FW- A	0.9 - 3.27	2.3 - 3.8	6.5 - 8.0	10	1.0	2.0	25
Sediments FW - B	1.83 - 18.58	3.0 - 7.1	7.2 - 11.3	15	1.0	2.0	25
Sediments FW- C	0.6 - 30.39	2.0 - 8.4	6.1 - 12.5	20 - 30	1.0	1.0 - 2.0	25
ORE	1.66 - 26.89	2.9 - 8.1	7.0 - 12.1	15	-	-	10 - 35
Tuff - HgW	9.82 - 79.93	5.5 - 12.0	8.0 - 15.0	-	0.5	0.5	25

16.2.3 Stope Ground Support Requirements

The development ground support requirements were assessed using empirical methods and are summarised in Table 16-3.

Table 16-3: Ground Support Requirements

Geotechnical Domain	Excavation Type	Minimum Ground Support Requirements (Range)
Footwall Sediments	Main Decline	2.4 m Galvanised Friction bolts (46 mm diameter Split Sets) Bolt spacing 1.0 m - 1.6 m Surface support ranges from Fibrecrete (75 mm) to Mesh covering backs and walls to grade line
Footwall Sediments	Level Accesses	2.4 m Galvanised Friction bolts (46 mm diameter Split Sets) Bolt spacing 1.0 m - 1.6 m Surface support ranges from Fibrecrete (75 mm) to Mesh covering backs and walls to grade line
Vein (Deposit)	Ore Development	2.4m Galvanised Friction bolts (46 mm diameter Split Sets) Bolt Spacing 1.4 m - 1.8 m Mesh support covering backs extending to grade line
Hanging wall Tuff Development	Other Capital Development	2.4 m Galvanised Friction bolts (46 mm diameter Split Sets) Bolt Spacing 1.5 m Fibrecrete (40 mm) to Mesh covering backs and walls to grade line

Crown Pillar assessment

Caving and potential subsidence was identified as a possible hazard during this study. An empirical assessment was completed to determine the stability of the crown pillar located between the top of the stoping and the base of the proposed open pit.

A number of mitigation measures are available and include limiting the number of stopes open at a given time and limiting stope widths to 15 m with a 10 m thick crown pillar. Alternately increasing the crown pillar thickness would allow for wider stopes in the transverse stoping area. Cemented backfill is recommended for the stoping area located immediately below the open pit and highly weathered zones.

Further investigation of the crown pillar coupled with numerical modelling would allow optimisation of the crown pillar thickness.

16.2.4 Stress

Stress measurements have not been completed for the Maud Creek project. A low to medium stress environment has been assumed for this study based on the depth below ground surface and expected regional stress regime. It is not expected that stress related instability will result from the proposed mining strategy. It is highly recommended that samples recovered from future drilling be sent away for Acoustic Emission (AE) stress measurement testing to confirm the stress field. Stress measurement testing should be followed by numerical stress modelling.

16.2.5 Backfill

Waste backfill is considered suitable for most areas within Maud Creek, where the mining method allows and sequence is bottom up. Cemented Rock Fill (CRF) will be required for strategic areas and sill pillar extractions as described above.

SRK experience at similar mines that utilise CRF backfill for sill pillar extractions suggest that a mix design of 7% cement (by weight) for stopes up to 10m wide is a reasonable assumption for this level of study. CRF with 4-5% cement has been used to create stable vertical exposures in preference to leaving mineralized pillars unmined. A detailed CRF mix design should be completed in future studies.

16.2.6 Further work

Recommendations regarding phase two work, summarised in Table 16-3, includes the following:

Dedicated geotechnical drilling program to:

- Target areas that require additional data such as the footwalls sediments, and crown pillar area;
- Drilling proposed locations of LOM infrastructure – portals, declines, vent shafts etc.;
- Improve understanding of structural characteristics including continuity and orientation variability;
- To collect representative samples for laboratory testing;

Mapping and photogrammetry in current pit;
Geotechnical laboratory testing using NATA certified laboratories;
Detailed backfill design for potential mining methods (including CRF);
Potential stress measurement testing (AE or DRA); and
Numerical modelling of proposed mining layout and sequence (including crown pillar and central pillar sequence)

Table 16-4: Drill program targets and estimated depths

Description	Drillholes (No.)	Depth (m)	Total (m)
To investigate sediments at depth	4	600	2,400
To supplement data and sediments across mid depths of mine	4	500	2,000
Portal and boxcut investigation drill holes	3	50	150
To investigate Infrastructure located in sediments (can be separated into multiple shorter drill holes)	1	600	600
Open Pit geotechnical holes – will provide data on crown pillar area and upper sediments	6	150	900
Total meters for proposed program			6,050

16.2.7 Open pit Slope Stability Analysis

The preliminary open pit design recommendations have been based upon the geotechnical domains summarised in Table 16-4.

Table 16-5: Preliminary Slope Design Parameters

Domain	Geotechnical Safety Berm Width (m)	Ramp Width (m)	Bench Height Cases (m)	Depth Range (m)	Bench Height (m)	Bench Face Angle	Bench Stack Height (m)	Bench Stack Angle	Inter-Ramp Angle	Limiting Overall Slope Angle
Hanging Wall	N/A	24	All	0 - 25	12.5	55°	25	46°	38°	
			12.5	25 - 50	12.5	55°	25	46°	38°	
				50 - 100		60°	50	45°	33°	34°
				50 - 150			75	44°	35°	36°
				50 - 250					36°	36°
			25	25 - 50	25	55°	25	55°	46°	
				50 - 100		60°	50	54°	41°	43°
				50 - 150			75	53°	40°	42°
				50 - 250					42°	43°

Domain	Geotechnical Safety Berm Width (m)	Ramp Width (m)	Bench Height Cases (m)	Depth Range (m)	Bench Height (m)	Bench Face Angle	Bench Stack Height (m)	Bench Stack Angle	Inter-Ramp Angle	Limiting Overall Slope Angle
Southern	15	24	All	0 - 25	12.5	55°	25	46°	38°	
			12.5	25 - 50	12.5	55°	25	46°	38°	
				50 - 100			50	42°	34°	35°
				50 - 150			75	41°	35°	36°
				50 - 250					34°	35°
			25	25 - 50	25	55°	25	55°	46°	
				50 - 100			50	50°	38°	39°
				50 - 150					40°	41°
							75	48°		
				50 - 250					39°	40°
Footwall and GCFZ	15	N/A	All	0 - 25	12.5	55°	25	46°	38°	
			12.5	25 - 50	12.5	55°	25	46°	38°	
				50 - 100			50	42°	35°	33°
				50 - 150			75	41°	36°	35°
				50 - 250						34°
			25	25 - 50	25	55°	25	55°	46°	
				50 - 100			50	50°	40°	37°
				50 - 150			75	48°	41°	40°
				50 - 250						39°

16.3 Open Pit Mining

Open pit and underground mine planning was conceptually undertaken to estimate the mining inventory of the project. The mine plan includes Inferred Resources and scoping study level assumptions supporting the modifying factors and design guidelines, hence the mining inventories from the mine planning do not meet should Mineral Reserves as defined by the NI 43-101 instrument.

16.3.1 Open Pit Optimisation

Open pit optimisation was used to identify the optimum economic pit shape based on the highest project cashflow. The pit optimisation process seeks a solution to a complex 3D mathematical relationship involving the Mineral Resource model, geotechnical slope guidelines, product revenue, project constraints, modifying factors and costs.

The key inputs into the optimisation process include:

- Overall pit slope angles based on geotechnical recommendations;
- Mining costs estimates;
- Mining dilution and mining recovery parameter estimates;
- Material re-handling and processing costs;
- Processing recoveries;
- Product price and revenues;
- Selling and transportation costs (including rail and port costs); and
- Royalties and corporate overheads.

The Mineral Resource model was converted to a mining model by a process of regularisation to account for a selective mining unit size of 10 m x 10 m x 5 m vertical. Mining dilution of 10% and production losses of an additional 10% were applied in the optimisation and downstream mine planning processes to account for the anticipated selective mining.

The inputs used in the optimisation process are presented in Table 16-1.

Table 16-6: Optimisation parameters

Description	Optimisation Input
Software Used	Whittle™ Optimisation Software
Pit Geotechnical Design Parameters	Overall slope of 37 degrees
Mining Dilution and Ore-loss Philosophy	Mining recovery 90% Mining dilution 10%
Mining costs	AUD4.25/t mined. Applied as a single cost to all benches. Cost is supported by provided client information
Processing costs	AUD58.4/t transitional and fresh including allowances for G&A and concentrate transport and handling. Based on 600kt/pa Processing Plant,
Gold Price	AUD1,450/oz
Processing recovery	95% Fresh 85% Transitional Oxide not considered as economic
Product transportation cost	Concentrate handling costs included in processing costs.
Resource categories considered	All mineralization including Measured, Indicated and Inferred as defined in the Mineral Resource model.
Underground interface	Underground interface sensitivity analysis considered 95% mining recovery and only considers fresh mineralization. Underground mining costs of AUD49.65/t applied.
Limitations Applied	A limit was placed on the open pit mining at 1015 mRL elevation to leave the crown pillar in lower grade material.

The refinement of the inputs was progressive, particularly around the underground to open pit interface. A maximum open pit mining depth was manually applied to the 1015 mRL elevation to limit the access of the open pit to the higher grade mineralization. This depth limitation ultimately had limited impact on the final pit shell selection as the non-elevation restricted pit shells did not extend significantly deeper.

The selected base case optimisation pit shell which considers the underground to open pit interface offered an inventory of 634 kt at a grade of 5.12 g/t Au containing 104 koz of gold.

A range of solutions were conducted to test the sensitivity of the project to changes in base assumptions. The project was found to be sensitive to the consideration of underground mining. The underground mining analysis assumed an underground mining cost of AUD49.65 /t and 95% underground mining recovery. Removing the underground option increased significantly increased the pit shell inventory to 600 kt at an average grade of 5.54 g/t Au containing 107 koz gold.

The optimisation considered no revenue recovery for the oxide mineralization. SRK notes that future consideration of processing oxide mineralization offers the potential for a reduced strip ratio and a larger open pit. Physical limitations with the interface between the underground mine may ultimately restrict the pit size and overarch a future opportunity to consider oxide processing.

The cut-off grades reported by the optimisation processes were 1.65 and 1.48 g/t Au for the Transitional mineralization and Fresh mineralization respectively.

The relatively high cut-off grades are due to the high total processing and transportation charges.

16.3.2 Mine Design

The base case optimisation pit shell was used to guide the open pit mine design. The design parameters, presented in Table 16-2 were used in the mine design.

Table 16-7: Open Pit Design Parameters

Description	Design Input
Site layout	Brown fields site with prior open pit workings
Pit Geotechnical Design Parameters	37 degrees used in the optimisation
Minimum Mining Width	10 m x 10 m
Internal pit ramp parameters	18 m double lane ramp, 9 m single lane
Staged design logic	Top-down development with no internal staging
Waste Dump parameters	18 m wide ramp, 20 degree batter slope, 10 m wide berm, 15 degree overall slope

The geotechnical mine design recommendations recommended overall pit slopes of between 34 and 38 degrees depending on the pit sector. The mine design utilised 55 degree batter slopes, 13.5 m wide berms, on a 25 m high bench to achieve a 45 degree inter-ramp angle. The measured slopes range between 30 and 38 degrees depending on the pit sector measured and the number of ramp intersections in the sector. As the open pit design is shallow and small, the ramp has a large impact on flattening the pit slopes.

The pit ramps were positioned to exit to the west of the operation in an alignment very similar to the existing as-built pit. This access aligns with the waste dump and RoM pad location concept layouts.

SRK lengthened the pit ramp in an effort to recover more of the targeted mineralization. In doing so, SRK utilised the existing operations ramp on the west and northern side of the existing pit to align the new pit ramp. This incurred waste material outside the pit shell but allowed the pit depth to increase. The resulting design depth of the pit was the 1078 mRL level compared to the base case pit shell reaching 1060 mRL. The net result is the open pit design has a mining inventory of 168 kt compared to the base case optimisation pit shell with 634 kt of mineralization.

16.3.3 Mine Production Scheduling

An open pit mining schedule was developed to demonstrate the open pit mine production over time. The pit is has an inventory of less than a year' s mill feed. The criteria used for the open pit mining schedule included:

- In situ Mineral Resource model used, scheduling all confident classifications of including measured, indicated and inferred resource categories;
- Cut-off grades applied to RoM mineralization;
- Oxide mineralization treated as waste;
- Quarterly periods;
- The mill production rate 600 ktpa and assumed constant;
- Top-down schedule with overburden mining in Quarter 1; and
- Total material movement targeting 1,000 kt per quarter

The open pit mining schedule was based on scheduling in situ materials using cut-off grades and the conversion of in situ mineralization to potential mill feed undertaken by including mining loss and dilution post scheduling.

Oxide mineralization was flagged in the production schedule to enable it to be independently reported.

Vertical rate of advance was not used as a scheduling constraint. The resultant schedule is aggressive with the 50 m deep pit being mined-out in 7 quarters. This is considered achievable if operational delays for grade control and pit dewatering are actively managed.

Salient points from the schedule include:

- Open pit mine life of 7 Quarters;
- Quarter 1, used for removing overburden and a cutting back around the existing pit;
- Quarter 2, commencement of production; and
- Quarter 3-7, lower strip ratio with the majority of the open pit mining in this period.

Table 16-8 shows the breakdown of the material movements by quarterly period. The reported information is based on in situ mineralization and subject to cut-off grades. All oxide mineralization is included in the waste totals in this report.

Table 16-8: In situ material scheduling

Period	Mineralization above cut-off grade		Waste Inventory ('000 t)
	Inventory ('000 t)	Grade (g/t Au)	
Quarter 1	0	0.0	900
Quarter 2	101	4.0	900
Quarter 3	143	5.2	800
Quarter 4	100	5.4	800
Quarter 5	100	5.4	800
Quarter 6	100	5.4	800
Quarter 7	89	5.4	0
Grand Total	**634**	**5.1**	**5,000**

A high level summary of the RoM production schedule is shown in Figure 16-3. Oxide mineralization has been reported separated. Transitional and fresh RoM mineralization has been subject to cut-off grades and mining loss and dilution estimation.

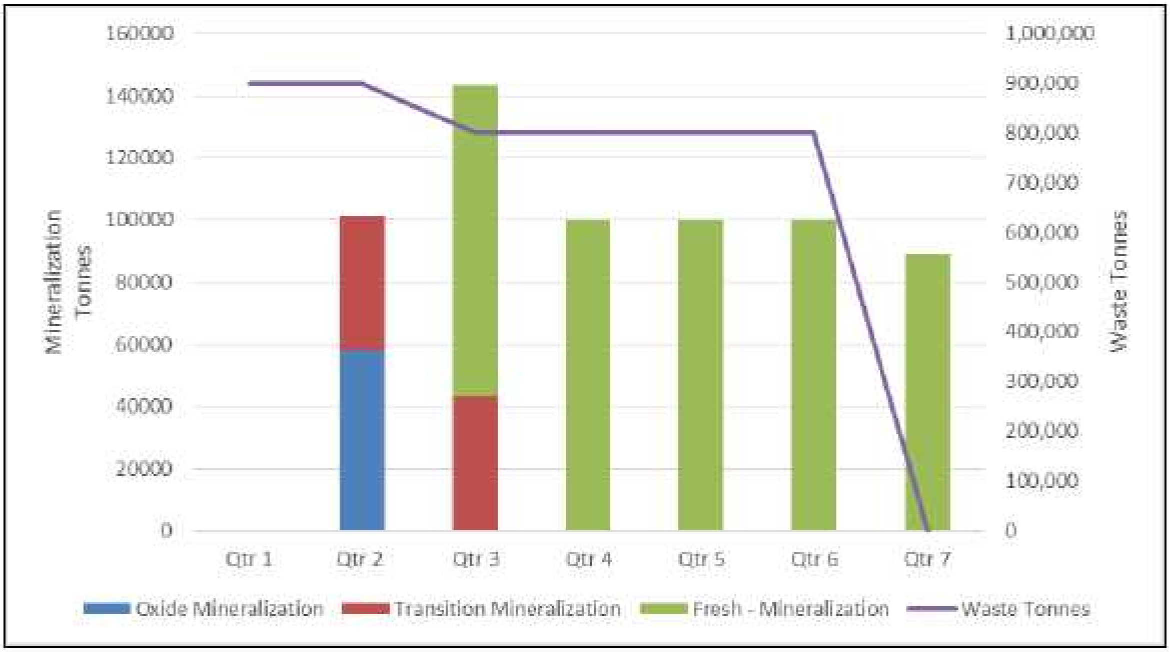


Figure 16-2: Total material movement

16.3.4 Waste and Stockpile Design

A waste dump was designed to the west of the open pit. The dump access ramp was aligned to minimize the haulage distance of waste from the pit to the waste dump. The design parameters and assumptions used for the waste dump are:

- 18 m wide double lane ramp;
- 20 degree batter slope;
- 10 m wide berms on 10 m high benches;
- 15 degree overall slope;
- Maximum total height of 20 m; and
- Standoff of 85 m from the waste dump to the pit crest.

The waste dump size was set by the following inputs and assumptions:

- The mine plan indicates open pit waste production of 5.0 Mt.
- In situ density of the open pit waste estimated at 2.4 t/m^3.
- The underground mine produces 722 kt of waste.
- Total waste production is estimated at 5.7 Mt.
- Assuming an average 30% swell factor for the truck compaction in the dumps, the loose density of the waste is estimated at 1.85 t/m^3.
- The surface water diversion works require 200,000 m^3 of waste which equates to 370,000 t.
- Underground mining requires the equivalent of 3 Mt of in situ waste, the balance of waste material is will be stowed in a waste dump.

16.3.5 Open Pit Mining Method

The open pit mining operation proposed is short term with a short operating life of 9 months. The mining equipment considered is small scale. A conventional 80 t class excavator would be ideal as the loading unit and can be matched to 85 t class rigid frame mine trucks for haulage. The length of haul is anticipated to vary from a 2 to 3 truck haul.

The productive mining fleet is anticipated to be supported by a combination of a water truck, grader and bulldozer. This support fleet will maintain the haul road, pit floor, waste dump and drill and blast pattern preparation. Other minor equipment such as IT loaders, support trucks and explosives trucks will support the drill and blast and mobile equipment maintenance activities.

The bench configuration is anticipated to be 5 m drilling benches, mined in 2 x 2.5 m flitches, with the material types being defined by mark-out tape and paint as designated by the site geologists. The grade control solution has not been defined at this stage of study but could use reverse circulation ahead of mining, blast hole sampling, and/or mapping and/or chip and channel sampling. Reverse circulation to full pit depth supported by mapping would be the preferred approach as this allows the bench plans to be rapidly developed and minimising grade control lost time to support the proposed fast vertical rate of advance.

Drill and blast is envisaged, recommend drill and blast to ensure productivity as free dig estimates are often optimistic. Drilling would be conducted by conventional top hammer rig with a single pass 5.0 m benches with 0.5 m sub drill. Wall control can be achieved using batter holes. Blasting is anticipated to primarily be performed using ANFO due to the dry conditions.

Pit dewatering is proposed to be managed using sumps and pontoon mounted pumps. Sumps will be progressively developed ahead of the bench mining to ensure dry mining conditions. The pontoon pumps will be used as required to dewater the sumps with the discharge water delivered to settlement dams prior to discharge.

16.3.6 Forward works

The open pit mine planning has been evaluated using a well-established, valid and appropriate approach. There are, however, a number of key assumptions in which the assessment is sensitive to and SRK recommends that these areas be subject to further definition as the study advances. These are outlined below:

- The open pit to underground interface and associated crown pillar has not been optimised. The open pit is currently giving preference to the underground mining operations by using relatively high mining recoveries in the interface section of the deposit. Future mine planning and optimisation of this interface is likely to increase the project value going forward.
- The open pit scale is sensitive to operating cost changes and further development of the project definition and cost estimation will help define a more robust open pit mine planning solution.
- Waste dump size. A consideration by SRK was minimisation of the project waste dumps need to be minimised. Larger open pits supported by revised future inputs and assumptions will generate larger waste dumps. Ideally a defined limitation on the size, footprint or otherwise could be stated and worked towards as a project target or limitation. .

16.4 Underground

16.4.1 Mine Access

Two options were initially considered for accessing the underground workings:

- Box cut from the surface; and
- Portal located in the pit.

Due to the size of the open pit and the scheduling requirement to access the underground early to ensure that there is production continuity, the in-pit portal option was not considered further.

The designed mine access is located to the north-east of the pit as shown in Figure 16-4.

Figure 16-3 and Figure 16-4 show the decline located in the footwall of the deposit and the decline is designed to a depth of, 500 m below surface, 650 mRL.

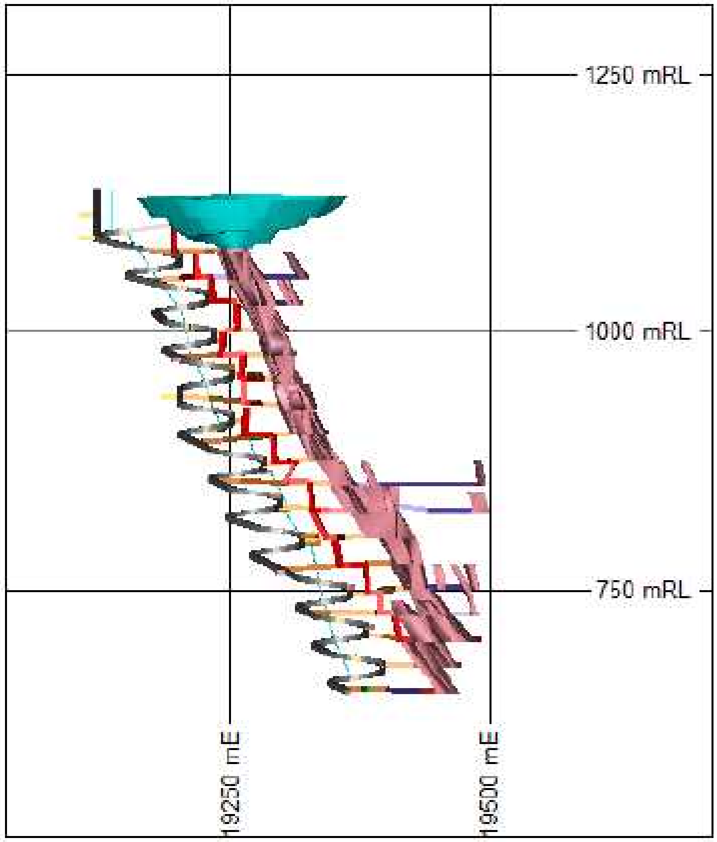


Figure 16-3: Decline Access – Section looking north

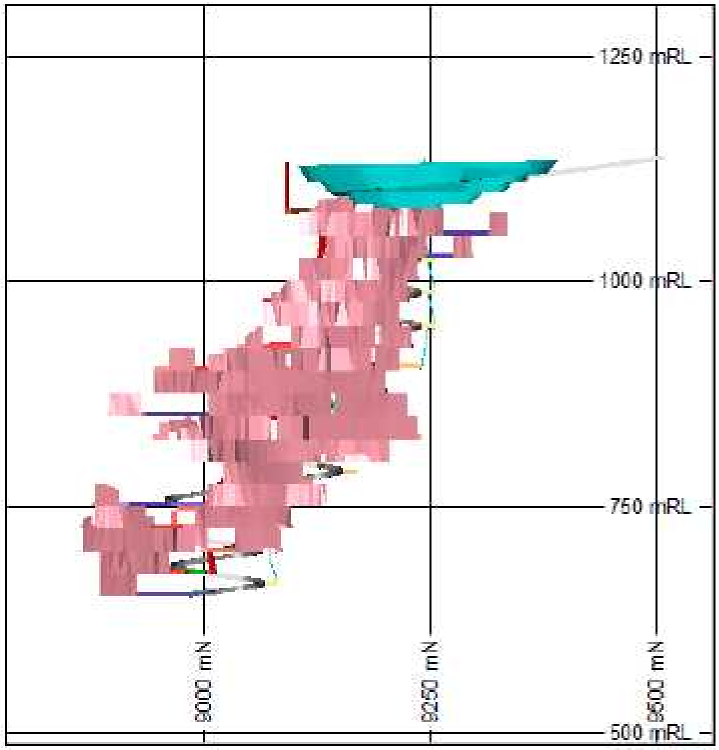


Figure 16-4: Long section looking west

16.4.2 Mining Methods

Underground mining methods can be categorized into three categories:

- Caving methods.
- Unsupported methods.
- Supported methods.

Given the type of mineralization and the width of the deposit caving methods are not appropriate for the project.

The unsupported and supported mining methods identified were appropriate to the deposit, based on the geotechnical review were:

- Long Hole Open Stoping; and
- Benching

These two methods are suitable given the deposit geometry and predicted ground conditions. Further design work and cost analysis determined that Open Stoping is the preferred mining method for the deposit due to the lower development requirements.

16.4.2.1Mining Method Description

The proposed open stope mining method has sublevel spacing of 25 m and stope strike lengths between 10 m and 30 m depending on the ground conditions and ore continuity. The mining sequence Figure 16-5 retreats along the ore-drive towards the central access and is a bottom-up method. Cemented rock fill will be placed in the stopes as backfill to allow the mining of the next stope in the sequence. Two mining front on each level are possible.

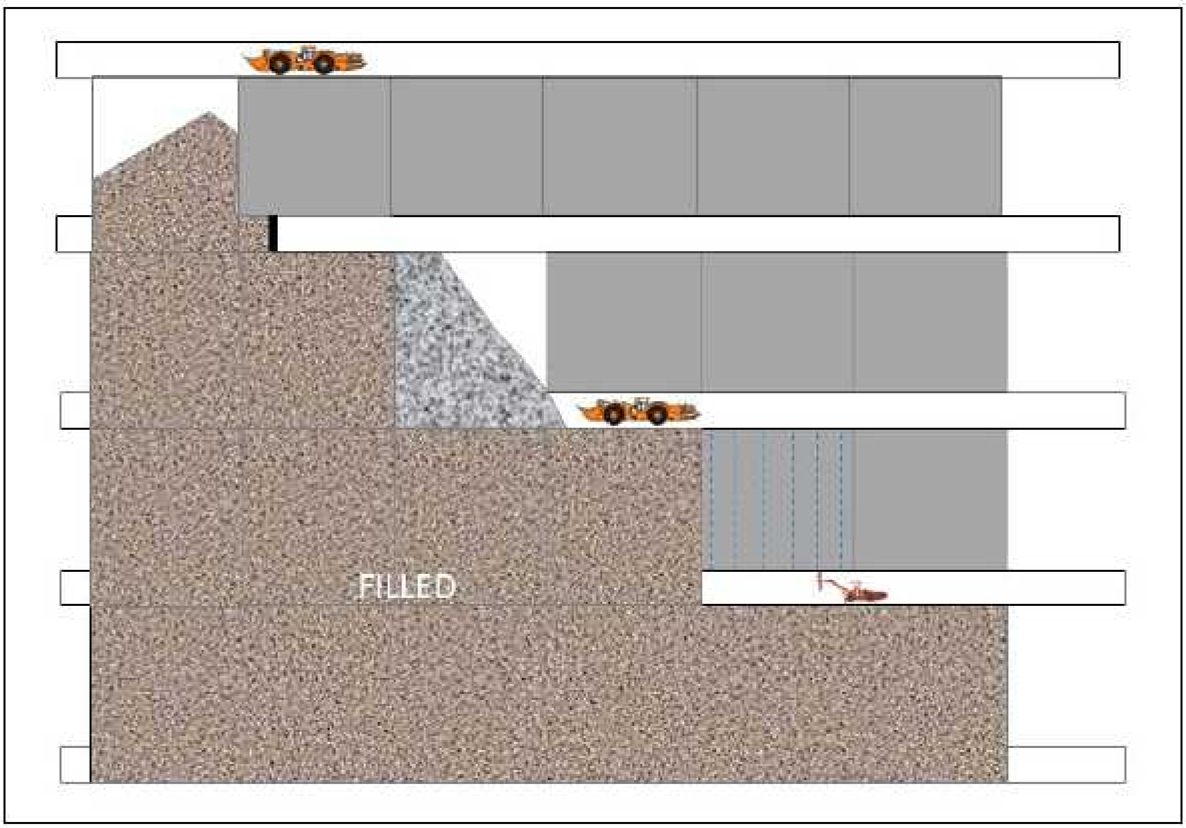


Figure 16-5: Long hole open stoping

16.4.3 Mine Design

16.4.4.1Cut-off Grade

Table 16-10 shows the cut-off grade inputs that have been applied for the Maud Creek underground design these resulted in a cut-off grade applied for the underground design of 2.65 g/t Au. This cut-off grade was applied as a “seed” value to identify the potential mining inventory.

Should the project progress, re-evaluation of the cut-off grade based on the findings of this study and future work is recommended.

Table 16-9: Cut-off Grade Inputs

Parameter	Unit	Value
Gold Recovery	%	92
Gold Concentrate Grade	g/t	60
Payable Gold	%	90
Gold Refining Charge	USD/oz	5
Gold Price	AUD/oz	1,415
Royalty	%	4
Exchange Rate	AUD: USD	0.85
Mining Cost	AUD/t	85.00
Processing Costs	AUD/t	25.00
General and Administration costs	AUD/t	15.00
Freight costs	AUD/oz	1.50

Figure 16-6 to Figure 16-8 show the sensitvity of the resource to the different design parameters that could be applied to the deposit at different cut-off grades.

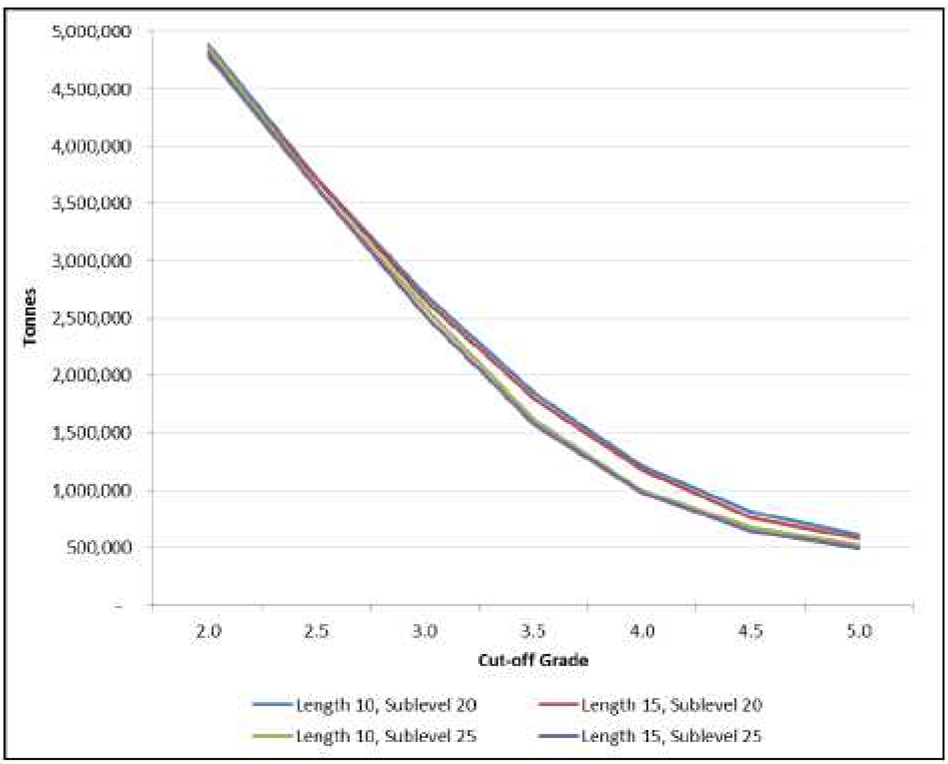


Figure 16-6: Cut-off Grade and Design Parameter Sensitivity – Tonnes

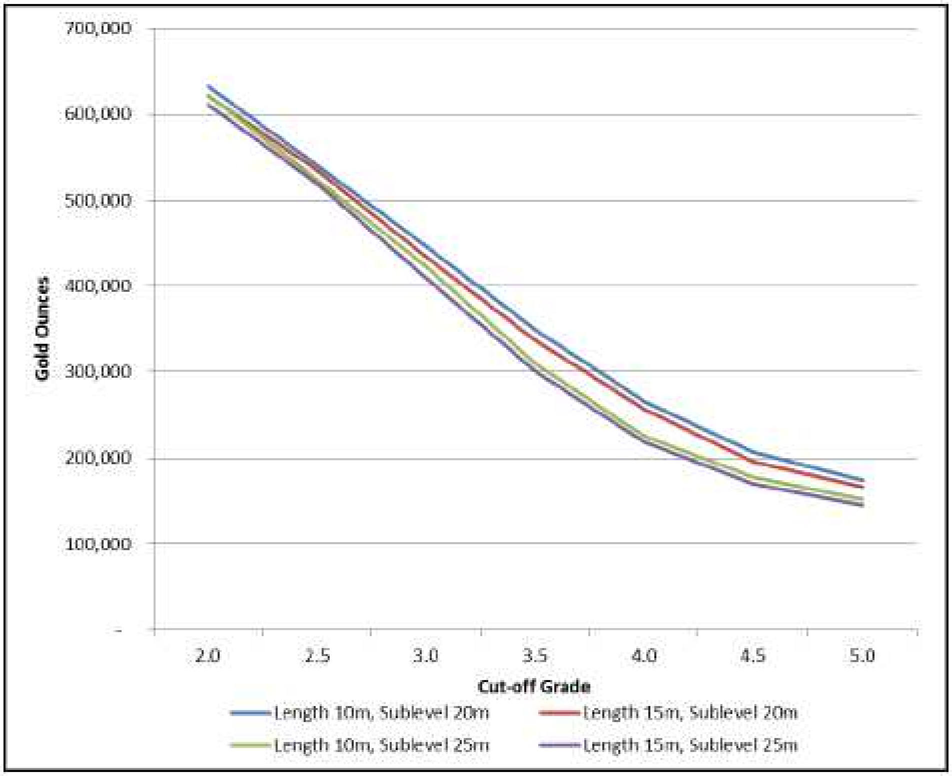


Figure 16-7: Cut-off Grade and Design Parameter Sensitivity - Ounces

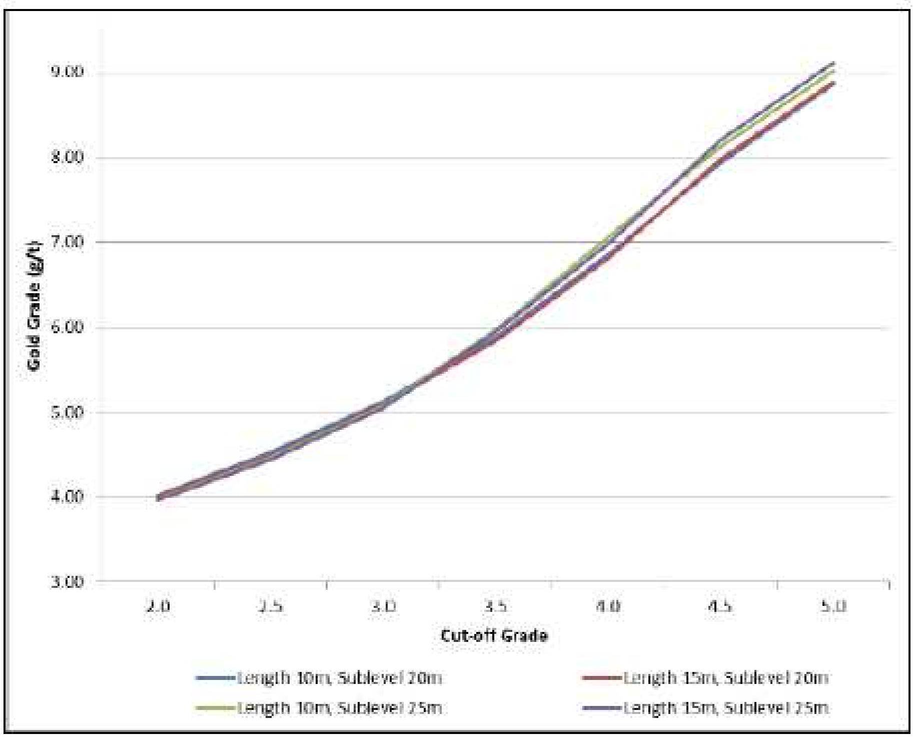


Figure 16-8: Cut-off Grade and Design Parameter Sensitivity - Grade

16.4.4.2Lateral Development

Each sublevel has an access drive, ventilation access and ore drives. The designed drive sizes are summarised in Table 16-11 and typical level presented in Figure 16-9.

Table 16-10: Lateral Development Design Parameters

Description	Size
Decline	5.5 mH x 5.5 mW
Stockpiles, Sumps	5.0 mH x 5.0 mW
Footwall Drive, Waste Drives	5.0 mH x 5.0 mW
Ventilation Access	5.0 mH x 5.0 mW
Ore Drives	5.0 mH x 5.0 mW

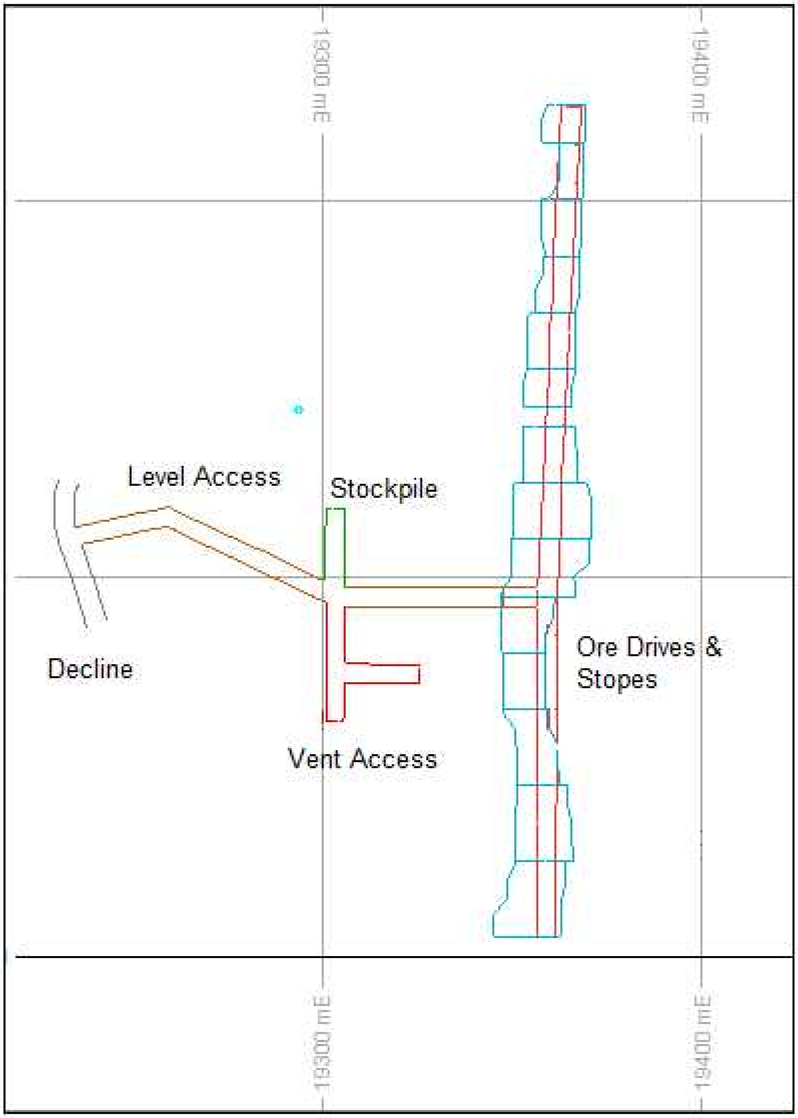


Figure 16-9: Typical sub level layout

The vertical development includes an escapeway and ventilation rises. Table 16-12presents the design parameters for the vertical development.

16.4.4.3Vertical Development

Table 16-11: Vertical Development Design Parameters

Description	Size
Return Air Rises	4.5 m x 4.5 m
Escapeway	3.0 m

16.4.4.4Ventilation Requirements

Fresh air will be drawn through the decline and escapeway rise and exhausted via the return rise system. The return air rise system is a series of 3 m x 3 m rises between levels to surface.

Total primary airflow requirement has been estimated based on 0.06m ^{3}/s for every kW power of mobile equipment according to Section 71.3 of the *Work Health and Safety (Mines) Regulation 2014* (NSW), which is the highest standard in Australia. The Northern Territory mining regulations do not provide any guidance. Table 16-13 shows the estimated peak production required airflow, and Table 16-14 presents the designed ventilation capacity of the mine.

Table 16-12: Required ventilation

Equipment	Power rating (kW)	No. of units	Total power (kW)	Total airflow (m^3/s)
Truck - 30t	293	3	879	53
LHD R1700	262	2	624	37
Service Vehicles	125	1	125	8
Light Vehicles	75	3	225	14
Total				**111**
Miscellaneous requirements (i.e. pump stations, explosive magazine, etc.)				**50**
System Losses (10%)				**16**
Mine ventilation requirements (with losses)				**177**

Table 16-13: Designed Intake and Return Air Capacity

Type	Description	Size	Capacity (m^3/s)
Intake	Decline	5.5 mH x 5.5 mW	182
	Escapeway	3.0 m	42.0
Total Intake			224
Friction Losses (10%)			22
Total intake capacity (with losses)			**202**
Exhaust	Return Air Rise	4.5 m x 4.5m	243
Friction Losses (10%)			24
Total exhaust capacity (with losses)			**219**

16.4.4.5Stoping

The stope design parameters are presented in Table 16-15. The parameters are based on the geotechnical domains and the cut-off grade from the preliminary cost estimates for the Project.

Table 16-14: Stope Design Parameters

Item	Value
Sublevel Spacing	25 m
Stope Length	10 - 30 m depending on ground conditions
Stope Width	4 -15 m
Cut-off grade	2.65 g/t

16.4.4.6Mine Service and Infrastructure

Water

Mine water will be supplied underground via the decline and service holes.

Dewatering of the underground mine will consist of a series of pump stations link via service holes, utilising the decline where required.

Compressed Air

Compressed air will be supplied underground from a surface compressor via the decline to each sublevel.

Power

The underground mine will require a 1000V power supply. The power will be distributed underground via the decline and service holes where required. The installation of underground substation has been allowed for in the design.

Emergency Egress

An emergency egress ladder way system is included in the mine design. Coupled with the decline, personnel will have a second means of egress from all sublevels within the mine.

During initial development it is recommended that refuge chambers be advanced along with the decline development face. Decline stockpiles can be converted to fixed refuge chambers as necessary to enable all personnel to be within close proximity of a refuge chamber or fresh air source.

16.4.4 Underground Dilution

There have been two types of dilution included in the Maud Creek design, planned and unplanned dilution. Figure 16-11 shows the definition of the dilution included in the Maud Creek mining inventory.

Planned dilution is the non-ore material (below cut-off grade) that lies within the designed stope boundaries (mining line). Unplanned dilution is additional material, which is derived from rock or backfill outside the designed stope boundaries. Unplanned dilution is predominately due to blast overbreak and sloughing of unstable walls. Unplanned open stope dilution is a measure of stope instability. Planned dilution can be controlled by optimizing the mining method and mining design.

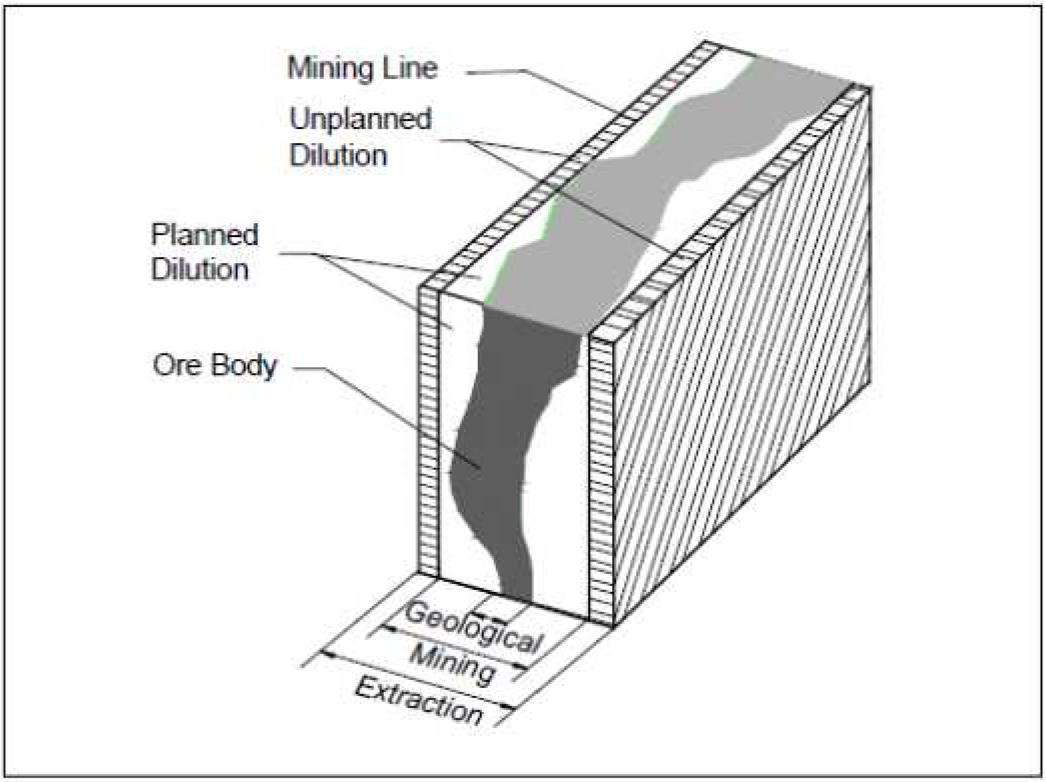


Figure 16-10: Dilution Definition
Source: A**fter Scoble & Moss, 1994)**

16.5.2.1Planned Dilution

The planned dilution is the dilution that is within the stope boundaries (mining lines, Figure 16-10). The range of the planned dilution for the mine design is summarised in Table 16-16. Where the planned dilution is greater than 15%, the stopes have been bulked out to the minimum mining width of 3 m.

Table 16-15: Range of planned dilution in mining inventory

Description	Value
Minimum	0%
Maximum	50.42%
Average	1.5%

16.5.2.2Unplanned Dilution

Unplanned dilution is the dilution that occurs beyond the stope boundaries (Figure 16-10). Typically, the factors that influence the unplanned dilution are shown in Figure 16-11.

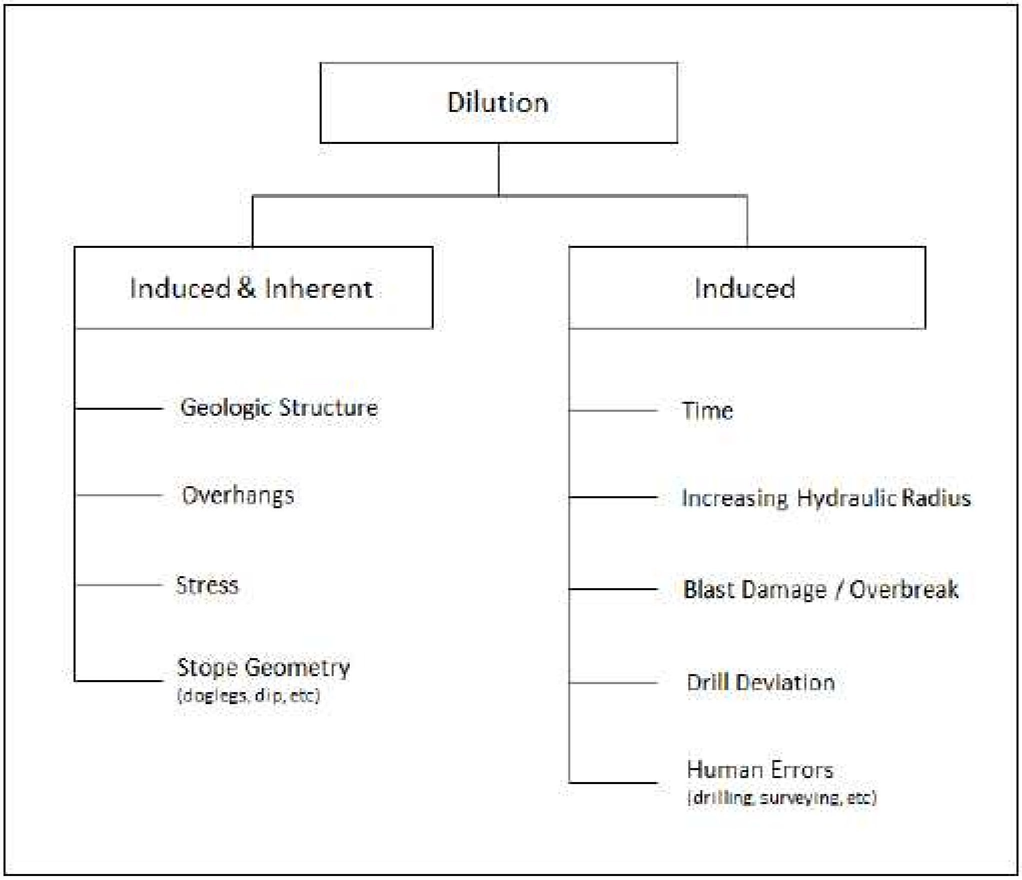


Figure 16-11: Influences on unplanned dilution

Source: After Pakalnis et al, 1995

The geotechnical assessment estimated that the equivalent linear overbreak slough (ELOS) for the hanging wall would be 0.5 m and the footwall 1.0 m. Using these figures the average dilution, 7% on the Hanging wall and 14% on the footwall has been applied to all stopes using a grade of 1 g/t Au.

16.4.5 Mining Recovery

The mining recovery has been calculated for the stope geometries designed, the average recovery has been calculated to be 95%. The calculations were based on the shape of the stopes and the expected material that will be unable to be loaded from the stopes. An angle of repose of 45 degrees has been assumed in these calculations.

Table 16-17 summarises the underground Mining Inventory for the Project by Resource classification, post application of dilution and mining recovery modifying factors. Figure 16-12 and Figure 16-13 present the tonnes and ounces by level.

Table 16-16: Underground Mining Inventory by Resource Classification

Resource Classification	% of Feed	Diluted Mined Tonnes and Grades		
		Inventory (kt)	Grade (g/t Au)	Contained Gold (koz)
Measured	20	679	6.1	132
Indicated	70	2,306	3.9	290
Inferred	10	291	2.8	26
Underground Mining Inventory	**100**	**3,276**	**4.0**	**423**

It is important to note that the PEA is preliminary in nature that it includes Inferred mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and there is certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have documented economic viability.

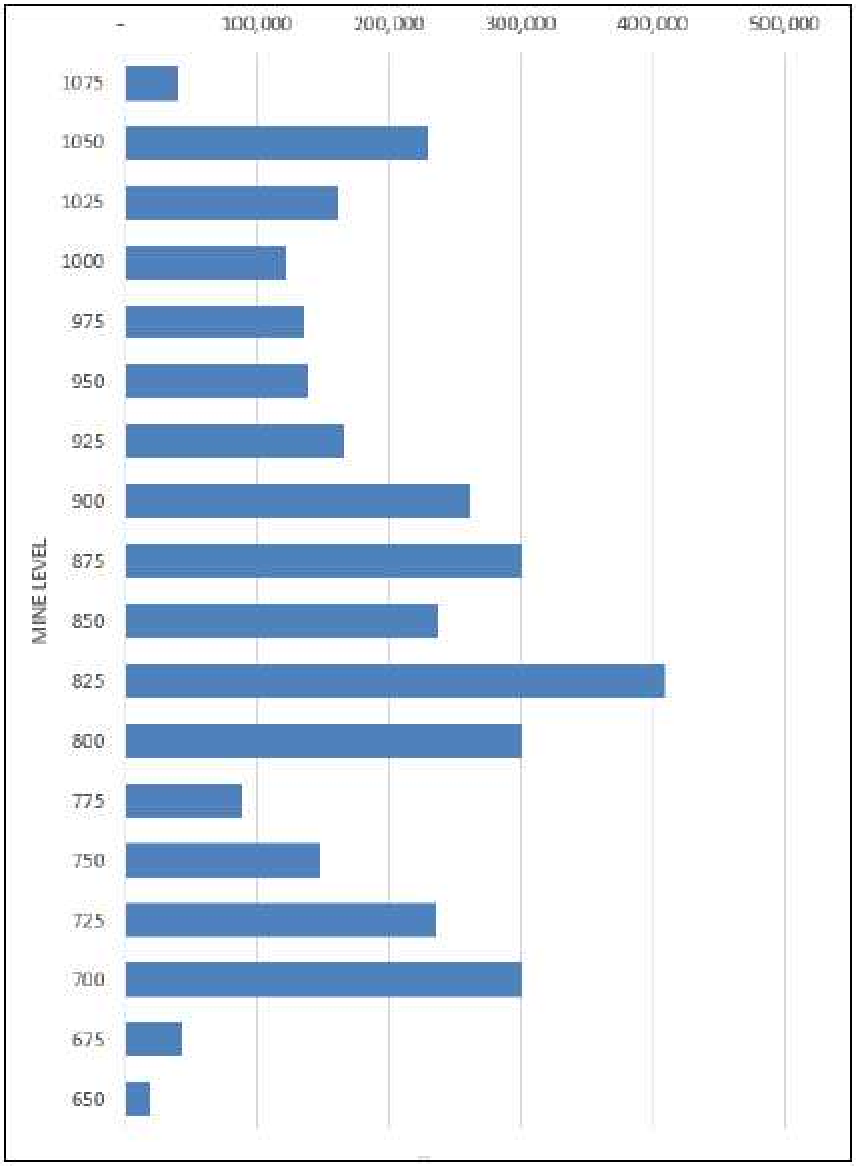


Figure 16-12: Chart showing production tonnes by Level

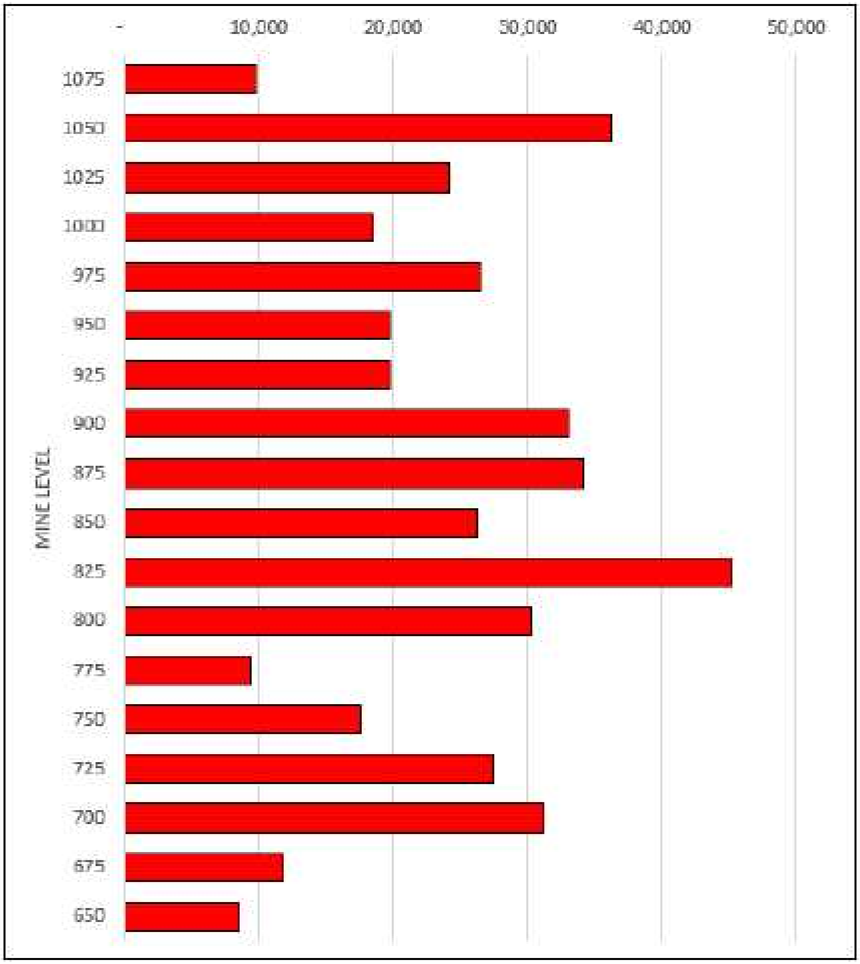


Figure 16-13: Chart Showing Ounces by Level

16.4.6 Underground Mine Schedule

The Maud Creek underground project schedule has the following key points:

- Mining activity is spread over a 6 year period;
- Steady state peak production target of 0.5 Mtpa ore production;
- First ore produced in Qtr 5;
- The production ramp-up period is 1.5 years, achieving peak production in Qtr 9;
- A maximum production rate of 0.5 Mtpa maintained for 5 years (average of 64 vertical m of production advance per annum);
- Simultaneous mining of multiple production fronts spanning multiple levels throughout the mine life will be crucial for maintaining a constant steady state ore production; and
- Production begins to ramp down after 4.5 years of full production because the number of active mining fronts begin to drop off as the known mining inventory is depleted.

Schedule Productivities

SRK has applied the productivities presented in Table 16-18 to the mining activities in the schedule.

Table 16-17: Typical productivities applied to the mining activities

Development	Units	Value
Single Heading		
Decline and other Capital Lateral Development	m/month	120
Operating Development	m/month	240
Shaft	m/day	3
Escape way	m/day	3
Rises	m/day	3
Multiple Heading		
Lateral Development capacity	m/month	60
Production Drilling	m/d	260
Stoping	t/d	840 (average per stope)
Backfill	m^3/d	1,900

Scheduling Strategy

The following scheduling strategy for Maud Creek deposit has been applied:

- Stope production to commence as soon as possible;
- A backfill crown pillar is created as shown in Figure 16- 14 to allow multiple advancing mining fronts at depth, this pillar is mined later in the schedule after the last stopes on the levels directly above and below have been filled;
- Development to be mined on a “just-in-time” basis;
- Return airways and escape way rises for each block to be completed prior to the commencement of stoping on each respective level;
- Scheduling of multiple production fronts over multiple levels are planned throughout the mine life are crucial for maintaining a constant steady state ore production.

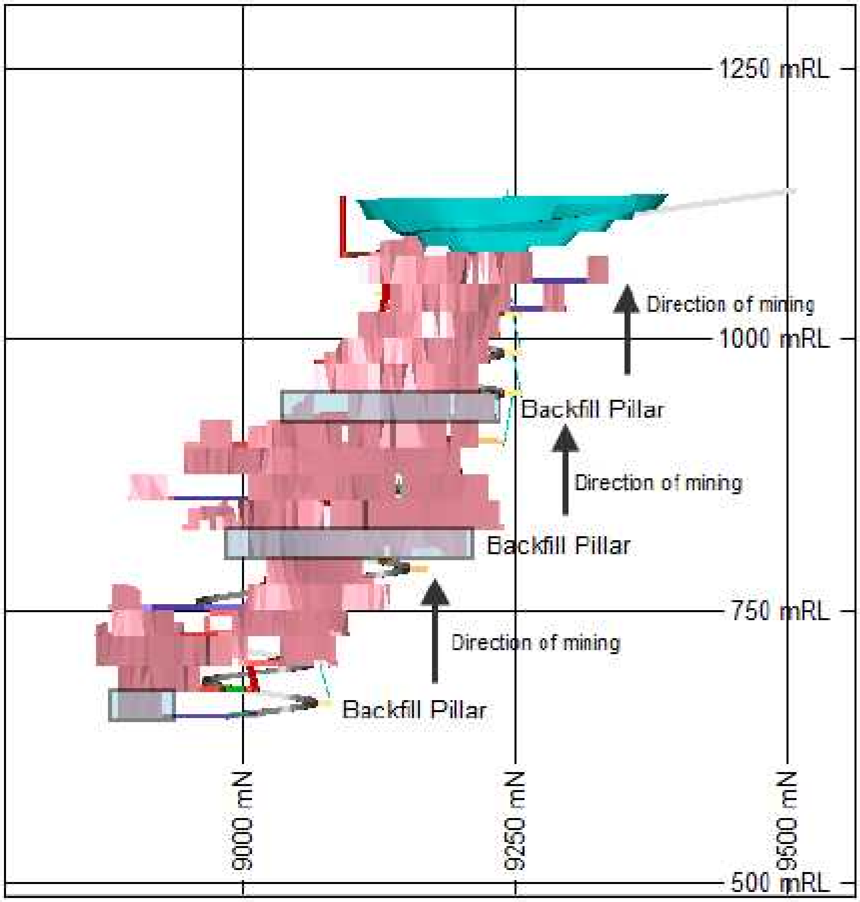


Figure 16-14: Long section looking west showing mining fronts

SRK applied Tatman' s (2001)[1] method to validate the suitability of the preliminary production rate for the Maud Creek deposit, shown in Table 16-18.

The average stope thickness varies between 4m and 35m in the transverse stopes. The average thickness is 10m. Using the table presented in Figure 16-15, SRK selected the deposit thickness row >10 m and the moderate risk column. The production rates resulting from the formula range from 0.25 Mtpa to 0.55 Mtpa. SRK selected a target production rate for scheduling of 0.5 Mtpa which falls within the moderate risk range and results in an average vertical m advance rate of approximately 64 vertical m per annum.

Table 16-18: Recommended rate multipliers by lode thickness

Deposit Thickness (m)	Low Risk	Moderate Risk	High Risk
>5 m	<20	20 to 50	>50
5 to 10 m	<50	50 to70	>70
>10m	<30	30 to 70	>70

[Annual production rate = Rate factor * Rate multiplier]

Rate factor = tonnes per vertical metre

Source: Tatman 2001

[1] Tatman, CA (2001). *Production-rate selection for steeply dipping tabular deposits,* Mining Engineering, *pp. 62-64,* October 2001.

Development Schedule

Figure 16-15 presents the quarterly target of approximately 450 capital development meters per quarter (104 per month) and 400 operating lateral development meters per quarter.

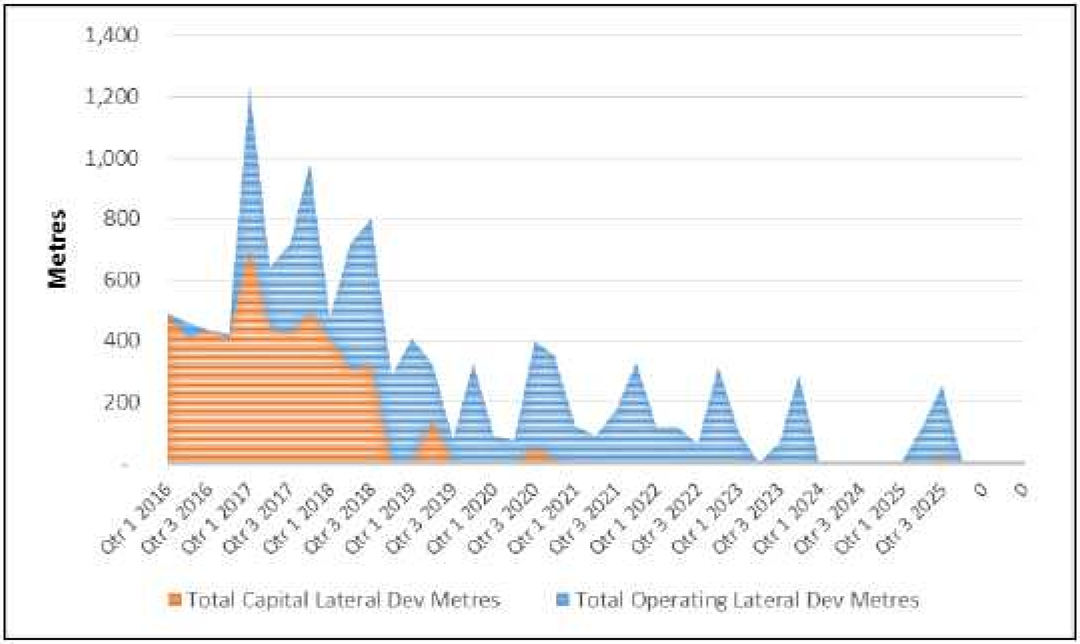


Figure 16-15: Development schedule

Underground Production Schedule

Figure 16-16 presents the underground production schedule and Figure 16-17 the ounces by quarter for the Project.

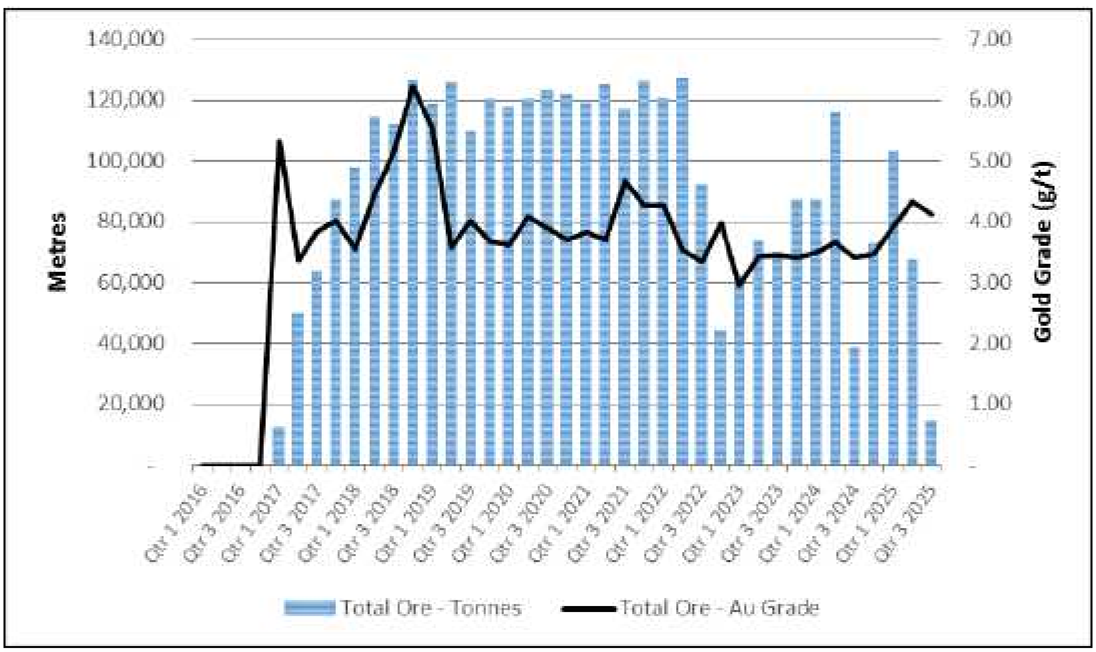


Figure 16-16: Proposed production tonnes and grade

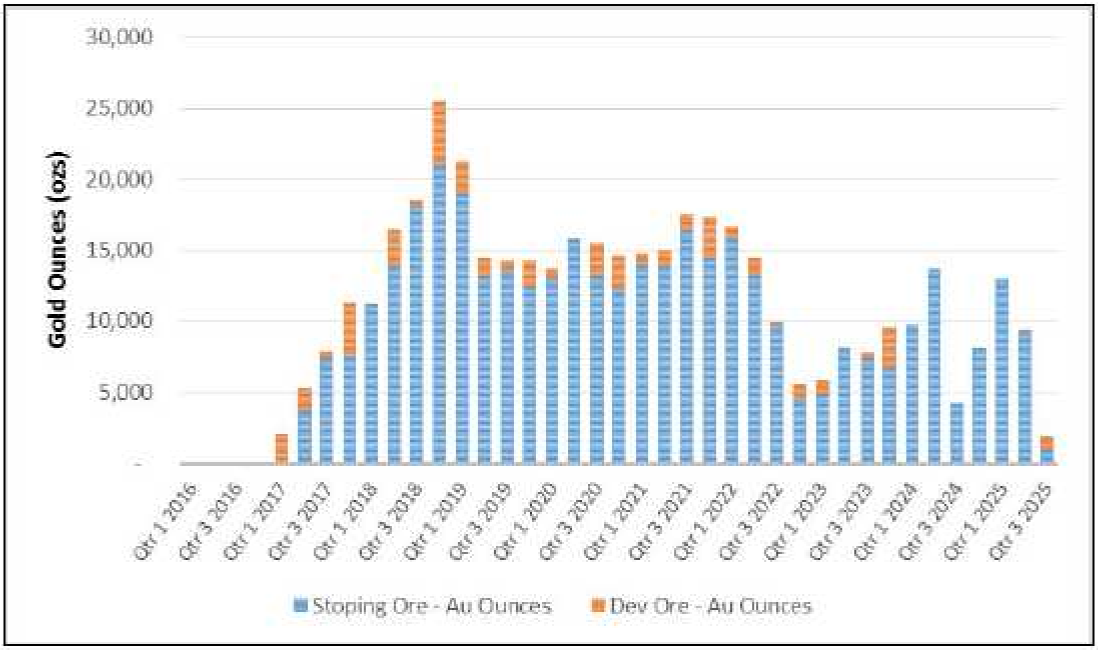


Figure 16-17: Gold Ounces by Quarter

Equipment Requirements

Table 16-19 summarises the equipment required to operate the underground. Allowance has been made for underground trucks, loader, drills and services vehicles.

Table 16-19: Equipment Requirements

Description	No. of Units
Trucks – 30t	3
Loaders -6 – 7 m^3 bucket	2
Development drills	2
Production drills	1
Cable bolter	1
Service vehicles	1
Light vehicles	3

Mobile Equipment

Table 16-20 lists the requirements for underground truck haulage; this is based on using 30 t underground haulage trucks. Table 16-21 lists the requirements for loaders, this is based on using 6-7 m^3 bucket loaders for both development and production requirements. Table 16-22 and Table 16-23 list the requirements for development and production drills.

Table 16-20: Trucks Requirements

Description	Units	Value
Peak time haul distance to surface	m	2,100
Average surface haul distance	m	400
Total travel time per cycle	min	24
Total loading time per cycle	min	5
Allowance for inefficiencies	min	5

Total cycle time	min	34

Description	Units	Value
Truck payload	t	30
Tonnes per hour per truck	t/hr	53
Average work hours per day	hr	18
Work days per year	day	365
Truck maintenance days per year	day	52
Effective work days per truck per year	day	313
Truck capacity per year	kt/year	298
Peak production (ore + waste)	kt/year	640
Required trucks at peak production	trucks	2.1
Total trucks required (rounded up)	**trucks**	**3**

Table 16-21: Loaders Requirements

Description	Units	Quantity
Loader production rate	t/day	1,000
Work days a year	day	360
Peak production rate (ore + waste)	kt/year	640
Required loaders	loaders	1.8
Total loaders required (rounded up)	**loaders**	**2**

Table 16-22: Required Development Drills

Description	Units	Quantity
Development drill rate		
Single heading	m/month	120
Multiple heading	m/month	240
Required development rate		
Single heading	m/month	104
Multiple heading	m/month	140
Required development drills		**2**

Table 16-23: Required Production Drills

Description	Units	Quantity
Drill density	t/m	12
Drill production rate	m/day	250
Work day per year	days	360
Peak stope production	kt/year	480
Required production drills	**drill**	**1**

Services Equipment

Table 16-24 lists the requirement for service vehicles, including service truck and light vehicles.

Table 16-24: Required Service Vehicles

Description	Number of units
Service Vehicles	1
Light Vehicles	3
Cabolter	1

16.5 Hydrogeology/ Dewatering

This section discusses the below-ground (Hydrogeology) aspects, the surface water management (Hydrology) aspects are discussed in Section 18.7.

SRK reviewed the findings and basis of previous reports and considers that them to be fit for purpose. Based on the absence of the underlying data, the purpose and stage of this study it was not appropriate to undertake additional hydrogeological modelling.

Should the study progress, additional data is required and should be incorporated with the revised structural / geological model to develop an updated numerical model. This would either build on the existing model, if data can be made available or a new model would be constructed. Specifically the updated model would confirm / address dewatering rate assumptions and would seek to collect data at depth. There is a distinct lack of knowledge at depth in the existing model.

16.5.1 Hydrogeology

Groundwater within the Maud Creek project area occurs within fractured, weathered and oxidised tuff, typically near the contact with main quartz vein. The quartz breccia zone located between the water table and around 100 mbs is interpreted to constitute a fractured rock aquifer. Groundwater may also occur at the contact and within the upper portion of the tuff (URS, 2007).

Groundwater recharge occurs by infiltration of rainfall. Flows in Gold Creek are also thought to be a possible source of groundwater recharge (URS, 2007).

The cavernous limestone of the Tindal Limestone aquifer to the west is likely to be hydraulically isolated from the fractured rocks and metasediments in the vicinity of the mine. Using test pumping data from two bores tested by Dames and Moore (1998), estimated values of transmissivity (T) ranged between 24 and 46 m^2/d. These values of T are considered relatively small and likely indicative of hard rock water bearing materials where fracturing is not well developed, poorly connected or both.

16.5.2 Dewatering

Previous Dewatering Analyses

Dames and Moore (1998) developed a groundwater flow model to determine the number of dewatering bores and pumping rates required to lower the water-table as part of planned open cut pit operation. The modelling found that seven dewatering bores with rates of between 300 and 500 m3/day (total dewatering rate of 2,200 m3/day) would be required to lower the water-table by 120m in an 18 month period (SRK, 2015).

URS (2007) expanded on the work of Dames and Moore (1998) by extending the model domain to simulate local and regional scale flow features. The model was used to predict dewatering of an existing open pit 200 m long by 100 m wide by 26 m deep and, in 2007, containing an estimated 300 ML of water (assumed mix of incident rainfall, minor runoff and groundwater inflow). The model extended to sufficient depth to assess the dewatering of underground mining to a depth of 700 m below surface (mbs) (-580 m RL). The model predicted dewatering rates of 3,400 m^3/day during the first year of mining, decreasing to 1,700 m^3/day after ten years of mining; this is similar to that proposed in this Technical Report.

Dewatering Assessment for Proposed Mine Development

Dewatering estimates for the proposed open pit and underground mine are based on the following assumptions and limitations:

The existing open pit excavation measures 200 m long by 100 m wide by 26 m deep and contains approximately 300 ML of water (mix of incident rainfall, minor runoff and shallow groundwater inflow).

The pit will need to be dewatered prior to underground mining.

The pit is within weathered and unweathered volcanics (tuffs).

The proposed open pit will be 300 m long, 200 m wide and 50 m deep and mined over a 9 month period.

Underground mining will commence from 50 mbs to about 500 mbs over a proposed 9 year period.

On-site hydraulic parameters are available from two pumping tests (in 1998) using bores which were drilled and completed to between 120 and 130 mbs and hydraulically connected to the various lithologies / aquifers over their entire depths. The reported parameters are considered reasonable for the lithologies cited given the paucity of actual field derived data.

There is no measured hydraulic data below 130 mbs. Noting the proposed mining development is expected to reach 500 mbs i.e. 370 m below the depth to which hydraulic parameters have been obtained.

Open Pit

Based on historical information and testwork, SRK estimated the groundwater inflows to the open pit to be 60 m^3/d (less than 1 L/s) based on the following relationships and assumed values:

$Q = \pi.^{K.h_o{}^{2} / \ln (r_o/r_{pit})}$

Where,

K = horizontal hydraulic conductivity (m/d) ‑ 0.06 m/d

h_o = height of water surface above pit/aquifer base (m) ‑ 25 m (assume only radial flow)

r_{pit} = radius of pit (m) ‑ 80 m

r_o =radius of zero drawdown (m) ‑ 530 m. In order to obtain ro the following relationship was used:

$r_o = (2.25. T.t/S)^{1/2}$

Where,

T = transmissivity (m2/d) ‑ 35 m2/d

t = time since discharge commenced (day) ‑ 90 days

S = storativity term (dimensionless) ‑ 0.025

Dames and Moore (1998) assessed that to dewater a pit of roughly similar horizontal dimensions as proposed in this Technical Report, seven dewatering bores pumping at a combined rate of 2,200 m^3/d (about 25 L/s) would be required to lower the water table to 120 mbs over 18 months.

Given that there is no recent aquifer hydraulic data, scaling the Dames and Moore (1998) dewatering rates to suit development of a 50 m deep pit is suggested. On this basis and assuming a linear relationship between pit depth and required dewatering rate, a pit to 50 mbs would need to be pumped at about 900 m^3/d (approximately 10 L/s) over an 18 month period (ignoring the stored volume in the pit).

Underground Mine

URS (2007) developed a numerical groundwater model for a pit and underground mine, with the model extending to about 700 mbs. The potential underground mine is within the dimensions of the URS (2007) model.

The outcomes of the model were that in the first year of underground development, pumping rates would need to be of the order of 3,400 m^3/d (40 L/s) declining to 1,700 m^3/d (20 L/s) over a ten year mine life. Initial pumping rates could be reduced if dewatering commenced prior to decline development.

Summary

The dewatering assessment is deemed an appropriate starting point given the stage of project development and that no additional groundwater hydraulic data has been obtained since the Dames and Moore (1998) work.

There is considerable uncertainty in the dewatering assessment given the paucity of aquifer hydraulic parameters and that the data is only applicable to depth intervals of 120 - 130 mbs, whereas underground development is proposed to extend to about 500 mbs.

The following needs to be addressed to minimise the risk to the project from uncertainties in the dewatering assessment.

- The representivity of aquifer parameters derived from only two pumping tests given any structural complexity in and around the pit and underground mine;
- The anticipated depth of the underground mine to 500 mbs means that the lower 370 m of the proposed mine has no representative groundwater hydraulic data; and
- It is uncertain as to whether structural elements have been incorporated in the existing models. This is essential as groundwater storage and flow will largely be controlled by structure in this geological environment.

16.5.3 Recommended future Study Work

Field Investigations

To reduce cost, additional work especially drilling should be multi-purpose. For example, any additional resource estimation / sterilisation drilling could incorporate some or all of the following:

- Air circulation drilling: noting of lost circulation zones; monitoring of discharge rate over a v-notch weir; monitoring of water quality (physico- chemical) with depth.
- Fluid drilling (diamond etc.): noting of lost circulation zones; downhole geophysical logging; packer testing.

A sub-set of holes could be completed as groundwater level monitoring bores using small diameter PVC casing to provide greater spatial and vertical coverage of the site and regional groundwater systems. Several monitoring bores exist across the project area and these may be suitable as observation bores (e.g. for possible pumping tests) in-addition to the function they currently serve as part of the groundwater quality monitoring network. Several test dewatering bores were installed by

Dames and Moore (1998), but the integrity and accessibility of these is unknown and should be assessed from bore condition surveys.

It is suggested that any deep holes be initially surveyed using Nuclear Magnetic Resonance (NMR) to provide downhole hydraulic data without the need for relatively expensive pumping or packer tests. The need for more detailed pumping or packer tests should be assessed based on the outcomes of the NMR surveys.

Installation of pressure transducer data-loggers in monitoring bores would be useful to assess relationships between rainfall and groundwater level trends as well as providing valuable data for hydraulic modelling calibration purposes.

Hydraulic Modelling

The following is suggested to improve the reliability of model predictions and thus minimise risks associated with inadequate dewatering knowledge.

- Incorporate updated geological modelling.
- Incorporate recent groundwater level monitoring data for improved calibration.
- Ensure structural complexity in the numerical groundwater model is sufficient to allow robust dewatering predictions.
- Incorporate any relevant downhole geophysical data acquired during field activities suitable for refining the numerical groundwater model.

The existing groundwater flow models should be used as the starting basis for any additional modelling, although this may require changes to the model geometry to reflect the current site and regional geological understanding. This assumes the existing groundwater flow models are readily accessible and importable into present versions of compatible software.

17 Recovery Methods

17.1 Background

Maud Creek sulphide (fresh) mineralization is refractory to direct cyanidation but respond well to flotation. There is significant but variable gravity-recoverable gold throughout. The gold is largely locked in sulphides and requires an oxidation process for downstream processing. Furthermore, the presence of preg-robbing carbonaceous material renders the mineralization 'double-refractory' .

The study and process design is focussed on the Maud Creek sulphide resource. However, the Maud Creek deposit also includes around 300 kt of oxide and 230 kt of transitional material. The oxide mineralization is generally amenable to direct cyanidation and will be treated through the existing Union Reefs Processing Plant. The transitional mineralization is less amenable to conventional cyanide leaching (is more variable) so alternatively, this transitional material may be treatable by flotation. Flotation recovery could be further improved following a sulphidization pre-treatment process. Controlled potential sulphidization (CPS) technology has been successfully applied on several gold and copper/gold mines, and may be beneficial at Maud Creek. Some preliminary testing would be worthwhile.

The main body of metallurgical testing and other development work was conducted from 1994 - 1998 for Kalmet Resources NL and Kilkenny Gold NL. Some minor work was done for Harmony Gold Operations Ltd in 2003, and then further testing and piloting were undertaken for Terra Gold Mining Ltd in 2006. Along the way there have been several engineering and consulting reports in support of the metallurgical development of the project. In particular, John W MacIntyre and Associates Pty Ltd supervised the earlier testing, starting in late 1996 and culminating in a detailed report: "Metallurgical Evaluation of the Maud Creek Project" in September 1998.

This review draws heavily on the MacIntyre report and the original metallurgical testwork reports, and also considers the later testwork where appropriate. The process design presented herein is based on the available information that is considered by SRK to be appropriate for this PEA. Some minor gaps remain and further development and optimisation work is recommended. Furthermore, the present mine plan is considerably different to that envisaged in the late 1990s so the original design recommendations must be treated with caution.

The process design philosophy is to add a simple, conventional, inexpensive and low-risk flotation circuit to the existing facilities at the Union Reefs Processing Plant, allowing flexibility with respect to circuit configuration and downstream processing. The base case presented here is to modify the Union Reefs Processing Plant to allow sulphide treatment (campaign treatment) by adding a flotation concentrator plant at the Union Reefs site, producing a gold sulphide concentrate for sale or possible in-house processing. The sale options are discussed briefly in Section 19, while the downstream treatment options considered in Section 17.7.

The Union Reefs Processing Plant currently treats free milling ores from the Cosmo underground mine by conventional crushing and grinding, gravity recovery and CIL processing. It has excess crushing capacity and two underutilised grinding mills at the current throughput rates on a hard underground mineralization. From 1999 to 2003, annual processing rates of 2.8 Mtpa were achieved, but it currently treats only ~750-850 ktpa, so it can handle the additional 500 ktpa of feed from Maud Creek. The addition of a flotation plant enables campaign treatment of sulphides and is flexible in respect to treatment of oxide and transitional material. The Union Reefs plant is currently operated on a 9 days on 5 days off roster, with the crushing circuit utilisation lower again and typically only with one of the two mills operating (the second mill is only required approximately 10% of the time).

There are a number of advantages associated with leveraging the existing Union Reefs Processing Plant rather than building a standalone Processing Plant at Maud Creek including;

- Lower processing and infrastructure capital costs;
- Lower first fill costs;
- Low additional sustaining capital costs;
- Simpler and quicker project implementation, reduced technical risk;
- Lower water demand at Maud Creek;
- Less onerous approvals for existing processing facility;
- Lower overall operating cost;
- Production creep opportunities with existing facility;
- Simple and cheap future expansion to meet any increase in mining production;
- Ability to process oxide mineralization, so increasing the LoM of Maud Creek;
- More flexibility in processing, including transitional mineralization being able to be processed through the cyanide leach or flotation circuits or possible both, (higher gold recovery);
- Extensive tailings storage capacity already available (lower capital cost); and
- Extend the LoM of the Union Reefs/Cosmo underground mine operation.

The key additional cost is haulage of mineralization. Other key considerations include the required approvals required for haulage and obtaining the social licence to operate.

17.2 Processing Plant Basis of Design

This study is based on historical testwork data and previous engineering studies provided by Newmarket Gold in consideration of the existing Union Reefs Processing Plant. The Project assumption is to process mineralization through a modified Union Reefs Processing Plant. A detailed integration study has not been undertaken into the Union Reefs processing option. This will be undertaken at the next Phase of the Study. However it is expected to be relatively simple, quick and straight forward. It presents an attractive opportunity.

Preliminary process design criteria (PDC) and process flow diagrams (PFDs) have been prepared as appropriate for this stage of study. Certain assumptions have been made based on incomplete knowledge of the metallurgy and market conditions. These assumptions are discussed in the 'Process Flowsheet Selection' section below. The typical engineering deliverables will be refined as part of future study work and will better reflect integration of Maud Creek mineralization into operations at Union Reefs.

A processing mass balance model for a standalone plant has been constructed showing flow rate/ tonnage and composition of all key streams. Equipment sizing and power draw calculations are performed within the model using the parameters specified in the PDC. The model inputs (PDC) and outputs are provided in the appendices.

The capital cost estimate has been built up from supplier estimates and the Simulus database for major mechanical equipment items, with estimates for minor equipment and fabricated items taken from the Simulus database. Mechanical equipment that already exists at the Union Reefs processing facility was then removed from the estimate. Earthworks, civil, structural, instrumentation and electrical disciplines have been factored from mechanical equipment costs. These factors have been adjusted to account for the existing site, systems, cleared areas, etc.

Other factored direct capital cost items include first fills, critical spares and warehouse inventory, laboratory equipment and supplies. Indirect costs are also factored. They include site temporary facilities, mobilisation and demobilisation, freight, vendor representation and site commissioning and EPCM. Other factored items already existing at Union Reefs such as site buildings, IT, communications and network equipment have been completely removed for these costs. A contingency of 10% has been included in the overall capital cost estimate. The capital cost estimate will be refined as part of ant future work with a focus on equipment sizing and costing and integration into the existing processing facility. It is likely there is still some double up of costs and that it can be reduced further because of the preliminary level of the costing.

Operating costs have been developed from first principles. Reagent consumption and grinding media rates are taken from the mass balance outputs and testwork. Unit costs are based on supplier quotes provided to Simulus Engineers by vendors during the previous six months. Labour requirements have been estimated based on the existing Union Reefs organisational structure assuming an extra shift of operators and maintenance is required as well as an extra flotation team member per shift. It uses recent industry assessment for salaries but will be updated at the next phase of study to use the actual salary structure at Union Reefs. Power consumption has been built up from the equipment list and power price based on the local grid prices. The operating cost estimate will be refined as part of future work. The bulk of the fixed costs have been removed as they are largely accounted for already through the Union Reefs operating costs. A future review may choose to share the fixed cost savings benefits between the Cosmo underground ores and Maud Creek mineralization.

The target accuracies for this stage of study work is +/-30% for capital cost and +/-20% for operating cost, based on the process design criteria as developed. There is potential for costs to rise or fall in future in line with future moves in exchange rates, equipment, material and labour prices.

17.2.1 Process Flowsheet Selection

Several earlier studies were used in conjunction with the testwork reports in as part of the process flowsheet selection. In particular:

- 1998 Signet Engineers “Process design Criteria 3081- G- 00-F-001 Rev A” (Draft)
- 1997 “Review of the Metallurgy, Capital Cost and Operating Cost for the Maud Creek Project”, Signet Engineering May 1997
- 1998 “Metallurgical Evaluation of the Maud Creek Project” by John MacIntyre and Associates Pty Ltd. Includes abridged process design criteria

Based on these reports as well as a review of the testwork demonstrating the poor response to conventional cyanide leaching, initially a simple single stage crushing and closed circuit SAG milling circuit, followed by centrifugal style gravity concentration, simple rougher / scavenger / scavenger cleaning flotation circuit followed by concentrate dewatering was chosen as the base case flowsheet. Once the decision was made to utilise the spare capacity at Union Reefs Processing Plant, the front end crushing and grinding circuit design reverted to the existing 3 stage crushing and closed milling circuit configuration existing at Union Reefs. The process overview diagram is shown below in Figure 17-1. It highlights the new areas required as part of the flotation circuit upgrade modifications. The preliminary plant layout is provided further below in Figure 17-3. Downstream processing options were only considered at a preliminary level.

The Union Reefs Process Plant was originally designed to process free milling but predominantly fresh rock ores from the Crosscourse Pit with a BBMWi of 16 - 18 kWh/t at a grind size of 106 microns. Annual processing rates of 2.8 Mtpa (~140 t/h Mill 1 and 190 t/h Mill 2 at ~300% recirculating load) were achieved between 1999 and 2003. These ores typically had an average gold head grade of 1.5g/t recovering 92 - 94%, of which 30% was recovered by gravity. Union Reefs is significantly oversized for the current feed from the Cosmo Underground Mine. The spare milling capacity presents the key opportunity to Newmarket Gold.

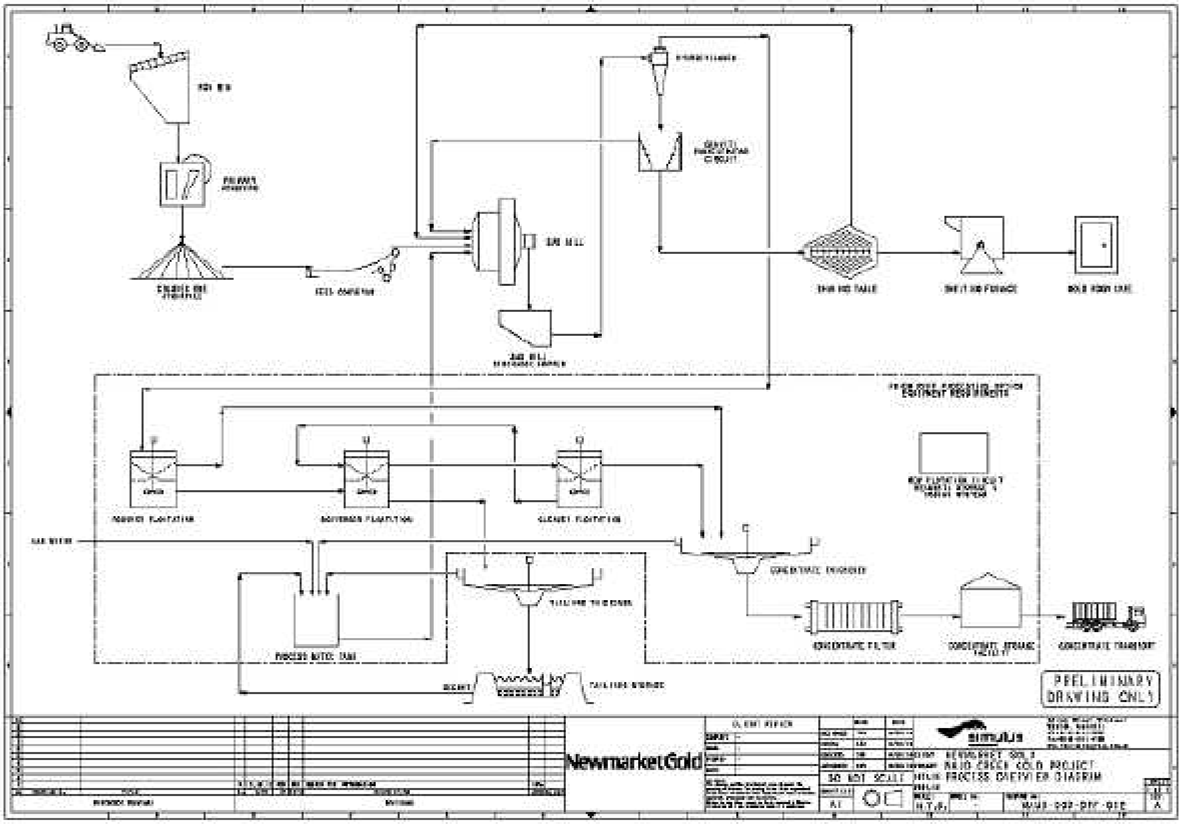


Figure 17-1: Maud Creek Gold Project Process Overview Diagram

At the Union Reefs Processing Plant, the ore is crushed and then milled in closed circuit with hydrocyclones. A separate gravity circuit recovers gravity recoverable gold from the ball mill discharge. The fine milled product from the cyclone overflow is pumped to a new flotation circuit where a high grade low tonnage gold bearing concentrate is produced from the sulphide minerals leaving a gangue non-sulphide residue that can be disposed of, to tailings. The concentrate is dewatered through thickening and filtration before being bagged, stored in shipping containers and transported by road, then ship to customers in China.

The selected grind size is 80% passing 75 microns. This was chosen following the 1996 optimisation testing, although earlier pilot programs used a coarser size, (125 microns in 1996 and 100 microns in the 1997 program). Further optimisation testing is recommended to confirm the grind size however the Union Reefs milling circuit will be flexible to a range of grind sizes.

The evaluation conducted by J MacIntyre in 1998 included a detailed economic optimisation of the grind size, as shown below in Figure 17-2.

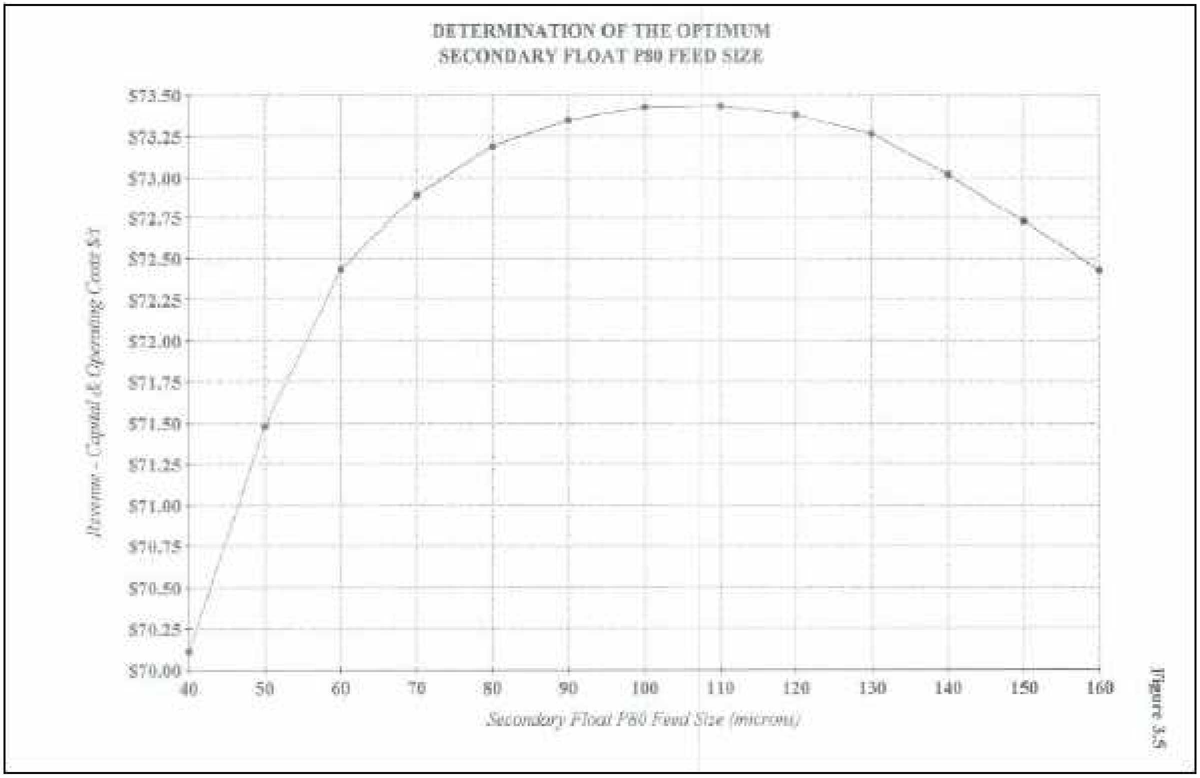


Figure 17-2: Economic Optimum Grind Size

Source: J MacIntyre, 1998

Although 100 - 110 microns was determined to be the optimum grind size, a P_{80} of 75 micron was judged more appropriate for design purposes. The slightly higher capital cost would be offset by better operating flexibility and lower risk.

Several important design parameters have changed since this evaluation was conducted:

- Circuit tonnage – was 300 ktpa now 500 ktpa
- Ore grade – was 7.4 g/t now approximately 4.38 g/t
- Comminution circuit remains three stage crushing + ball milling
- Downstream processing – was Bio-oxidation now direct concentrate sale
- Gold price – was AUD482/oz now >AUD1,500
- Power cost – was AUD0.125/kWh, now AUD0.22/kWh
- At that time concentrate was expected to be processed locally. With the base case of processing the concentrate at a remote location and the subsequent impact of concentrate transportation costs, a finer grind is further supported (higher concentrate grade and reduced mass pull).

The gold price appears to be a significant change. A higher price should mean a finer economic grind size, all other things being equal. Therefore there is unlikely to be a strong case for a coarser grind. On the other hand, recovery does not increase much at finer sizes. Grinding from 80 - 40 microns would increase gold recovery by about 0.8%, according to the projection model used by MacIntyre.

At this stage it is recommended to leave the design grind size at 75 microns. Further economic optimisation is recommended in the later stages of design. Processing through the Union Reefs milling circuit will be flexible to a range of grind sizes and this argument is largely redundant for design purposes. It can be optimised during operations.

The grinding circuit includes gravity gold recovery on a separate mill discharge pump. Gravity testing gave mixed results and the impact on overall recovery is not well established. However it does insure against coarse gold losses and enables on-site bullion production via smelting of the gravity gold concentrate.

The selected flotation circuit includes rougher, scavenger and scavenger cleaner stages, but no flash flotation. Some of the testing showed marginal benefit from flash flotation. It was found that the existing roughers in combination with the gravity circuit were highly effective, and addition of flash flotation is considered an unnecessary complication. However a flash flotation cell(s) could be incorporated if required. It will be considered further at the next stage of study.

A cleaner flotation stage will increase concentrate grade but possibly marginally reduce recovery. The economic trade-off will be affected by the choice of downstream processing route chosen, or direct concentrate sale. The cleaner circuit is included in the design as per the pilot plants, so product grade projections are consistent with the test results. It may well be possible to achieve the target 45g/t gold in concentrate without a cleaner circuit, but this circuit is fairly cheap in both operating and capital cost. Regrind of the scavenger concentrate might also be considered but has not been tested at this stage.

The flotation reagent scheme consists of sodium isobutyl xanthate (SIBX, collector), copper sulphate ($CuSO_4$, activator) and frother, with no pH adjustment. This scheme is commonly applied in flotation of gold bearing sulphide mineralization.

Given the high combined flotation and gravity recovery of gold, leaching of the flotation tails is unlikely to be economically viable. Earlier work recommended a tails leach circuit, but this was in the context of an integrated process plant including oxidation and cyanide leach of concentrate. With processing through Union Reefs, this operational flexibility is now an option to improve recoveries on transitional mineralization. Flotation tails can therefore report either to the CIL circuit or directly to tailings storage (via a tailings thickener due to water balance and environmental discharge considerations).

The gold flotation concentrate is thickened and filtered for transport. The gold is refractory and requires some form of oxidative treatment. The carbon content also creates preg-robbing characteristics. Circuit selection for the downstream process depends on a number of issues such as:

- Concentrate mineralogy, especially sulphur, arsenic, carbon and gold content
- Local economic factors, e.g. cost of power, reagents
- Environmental regulations
- Technology risk profile.

Downstream processing is outside the scope of this study. Some discussion of direct concentrate sale and other alternative downstream processing options available is provided in Section 17.7.

17.2.2 Plant Throughput Selection

Both the mining inventory and life of mine (LoM) need to be considered to give a reasonable plant throughput, with the plant sized initially largely driven by the desired life of mine. This in turn must be balanced against the operating and capital cost economies of scale and any financing obligations.

The preliminary underground mining inventory used for the plant sizing is approximately 2.5 Mt at 4.4 g/t Au, and the open pit mining inventory is 230 kt at 5.6 g/t Au. Both of these values are based on a cut-off grade of 1.5 g/t. While there is potential to grow the mining inventory, the plant sizing was undertaken based on the tonnes currently available at the time. This sizing now applies to just the flotation plant. SRK notes that significant increases in flotation capacity can be achieved with very low incremental increases in capital cost. If underground mining production is able to be increased, this provides a good opportunity to improve the project returns.

Based on a five year LoM, a 500 ktpa plant is recommended as the base case. A 500 ktpa plant size strikes a good balance between the current mining inventory, potential LoM extension, capital and operating cost increases, while maintaining a lower cost modular plant supply and construction philosophy.

The 500 ktpa throughput rate is further supported by the likely underground mining rates. Since the deposit is relatively narrow, a 500 ktpa production rate is a reasonable assumption. A higher mining rate would be challenging given the likely method and geometry of the deposit. The process plant can be simply debottlenecked (for example equipment at this throughput is often oversized) to achieve additional throughput if mining rates and LoM allow it in the future; this will be assessed in more detail in the full feasibility study. This LoM also allows the owner to economically amortise the The key process design criteria used for this PEA is shown in Table 17-1. Justification and explanations are provided through the preceding and following sections.

Table 17-1: Key Process Design Criteria Information

Variable	Unit	Value
LoM tonnes	Mt	3.8
Mineralization type	#	96% Fresh, 4% Oxide/ Transitional
Mill Tonnes	tpa	500,000
Plant utilisation	%	90
Milling throughput	tpa	70
Gold Feed Grade	g/t	4.38
Gravity Recovery (fresh)	%	20
Flotation Recovery (fresh)	%	75
Total Recovery (fresh)	%	95
Total Recovery (oxide/ transitional)	%	85
Recovered Gold	oz/a	66,890
Concentrate Mass Recovery	%	7.3
Concentrate Production	tpa	36,500
Concentrate Gold grade	g/t	45
Concentrate moisture	%	10
Tailings	tpa	463,514
Water Consumption (plant)	GL/a	0.35

17.3 Engineering Deliverables

A number of the key process engineering deliverables have been completed at a preliminary level for the Project. The deliverables were derived in two phases; the first phase involved engineering associated with a standalone plant located at Maud Creek and the second phase made allowance for integration and modifications to the existing Processing Plant at Union Reefs that is now the Project base case assumption.

All engineering deliverables would be updated during future phases of work.

17.3.1 Processing Description

The Processing Plant description is provided below along with some indicative drawings of the plant and the proposed modifications. The main stages of the process consist of:

- Three stage crushing and product screening circuit (existing)
- Two closed circuit ball mills, each with a hydrocyclone (existing)
- Gravity circuit including Knelson concentrators (existing)
- Flotation circuit (new)
- Concentrate thickening and filtration circuit (new)
- Concentrate storage and loading (new)
- Tailings thickener (existing)
- Utilities (largely existing)
- Reagents (largely existing).

It is expected that there will be some additional overlap as well as some additional modifications required to the existing plant. Confirmation is required to whether a separate process water circuit is required due contamination of the CIL circuit with flotation chemicals and vice versa. For example cyanide is often used as a depressant in pyrite flotation and this could impact gold recovery. It has been allowed for in the capital cost estimate. A plan drawing of the overall Processing Plant and Union Reefs with the proposed location of the flotation circuit is shown below in Figure 17-3.

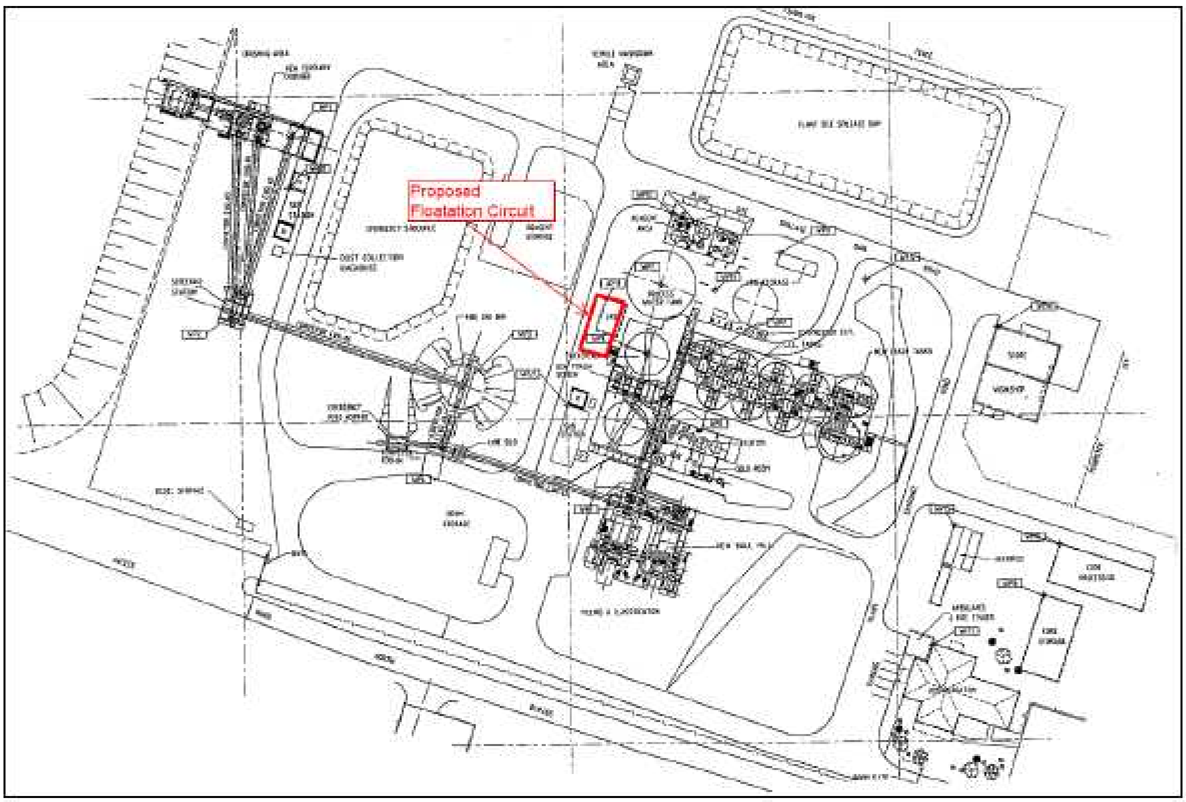


Figure 17-3: Union Reefs Site Plan

17.3.2 Crushing and Screening Circuit

The Unions Reefs crushing circuit is shown diagrammatically in Figure 17-4.

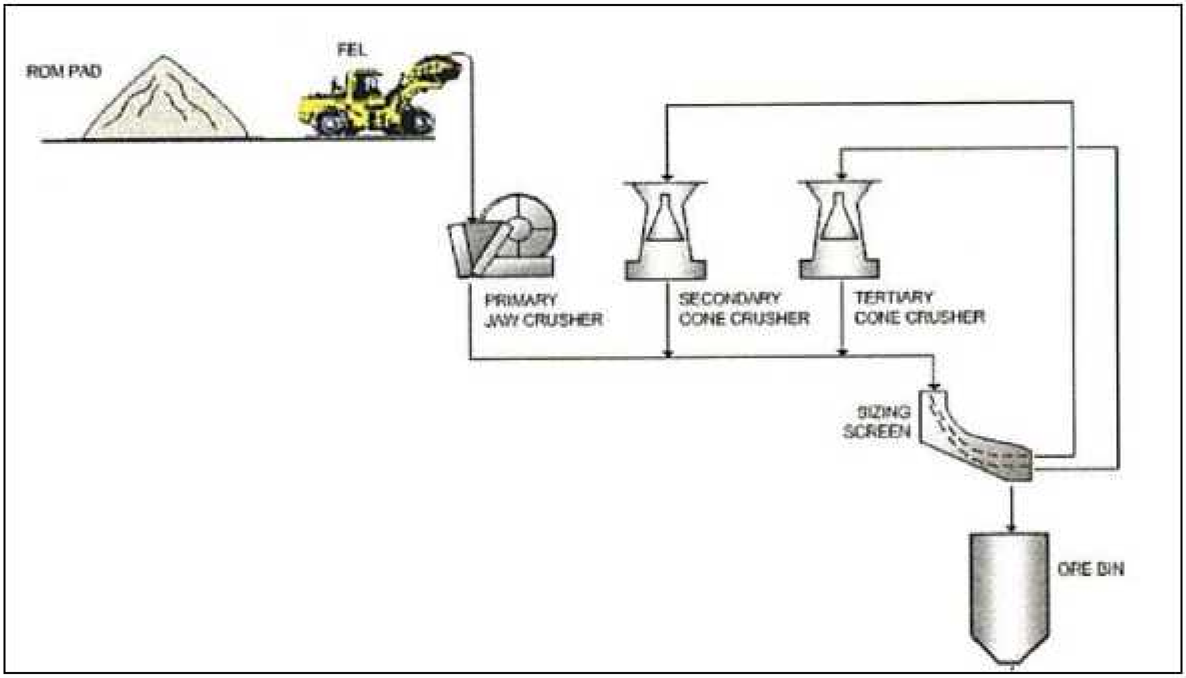


Figure 17-4: Crushing Circuit at Union Reefs Processing Plant

Run of Mine material is reclaimed from stockpiles by a wheel loader and tipped into a ROM bin. A vibrating feeder transports material from the ROM bin to a C140 Nordberg Single Toggle Jaw Crusher. Any oversize rock is broken at the Jaw Crusher by a hydraulic rock breaker. Jaw Crusher product is transported to a double deck "banana" 3.1 m x 7 m Nordberg Product Screen with a 40 mm aperture top deck and a 14 mm aperture bottom deck by conveyor belts. Tramp steel is removed after the jaw crusher by a fixed magnet to protect the secondary cone crusher and conveyor belts. Screen undersize (minus 14 mm) product is conveyed to a Fine Ore Bin/ Stockpile of 3000 tonnes live capacity, while screen oversize (plus 40 mm product) is transported to a Nordberg Omni-Cone 1560 secondary cone crusher. Secondary crusher product returns to the Product Screen. Intermediate product of minus 40 mm plus 14 mm from the screen is transported to a Nordberg HP500SX tertiary cone crusher. Tertiary crusher product is also returned to the product screen. No changes are envisaged to the existing crushing circuit.

17.3.3 Milling Circuit

Fine crushed ore is reclaimed from the fine ore bin by a slot type belt feeder. Stockpiled fine ore from outside the bin can be reclaimed using a wheel loader which tips to a day (emergency) feeder bin. Both the slot feeder and day feeder discharge to a single mill feed conveyor. The mill feed conveyor discharges to a Mill Feed hopper. The Mill Feed hopper has a split discharge onto two variable speed ball mill feeder conveyors, one feeding the 3 MW, 4.7 metre by 8.2 metre ANI No.1 Ball Mill, the other feeding the 4 MW 5 m x 9.1 m ANI No.2 Ball Mill. Each mill is able to be operated independently and/or in isolation, and each is in closed circuit with separate hydrocyclone clusters.

Mill discharge slurry overflows through a trommel screen into a discharge hopper, from where a portion is pumped to the hydrocyclones, a portion to the centrifugal gravity concentrator and the oversize scats material reports back to the ball mill.

Cyclone underflow is returned to the ball mill feed chute while the cyclone overflow reports to the flotation feed tank. The milling circuit at Unions Reefs is shown diagrammatically in Figure 17-5.

No changes to the existing milling circuit are envisaged.

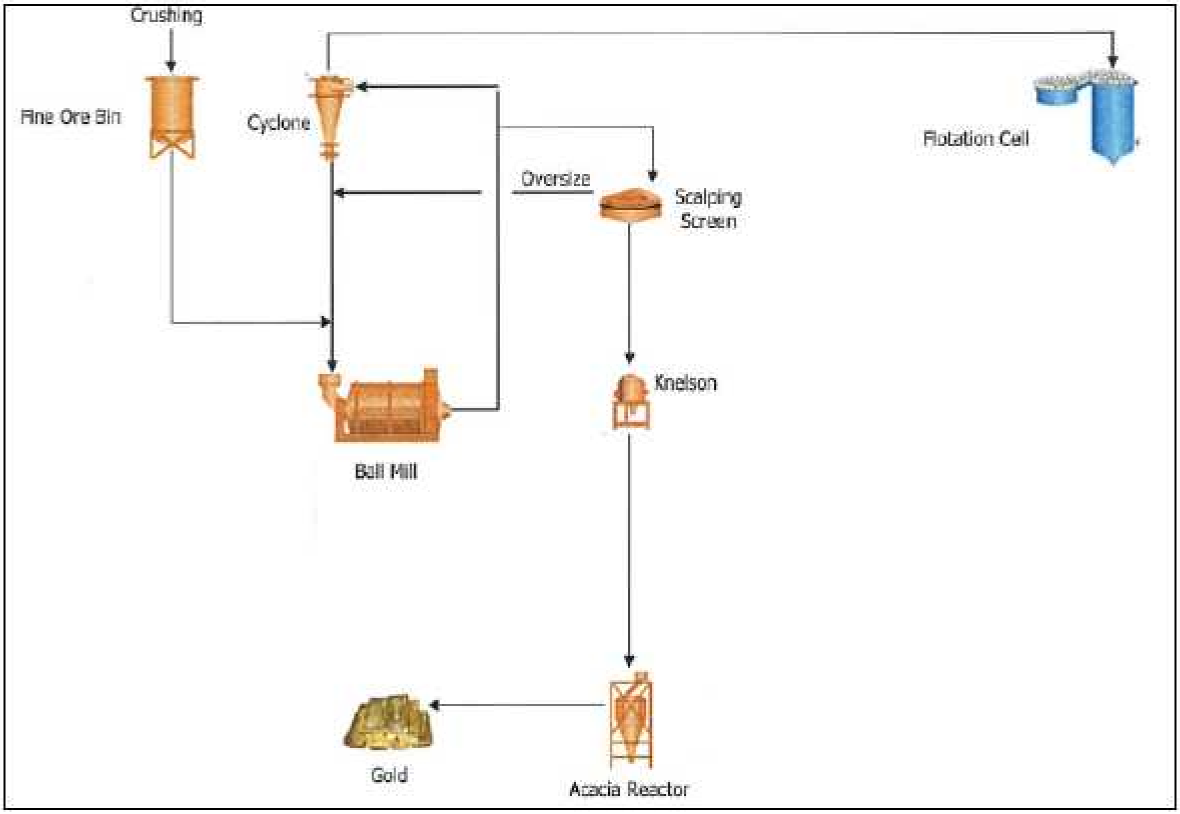


Figure 17-5: Grinding and Gravity Circuits at Union Reefs Processing Plant

17.3.4 Gravity Recovery Circuit

Each Ball Mill hydrocyclone cluster is fitted with a direct off-take on the hydrocyclone feed pot, which directs a bleed stream of ball mill discharge to a Nordberg 1.2 metre by 1.5 metre scalping screen to remove coarse scats. Fine screen product reports to one of two automatic discharge 30 inch Knelson concentrators. Rough Knelson concentrate containing coarse gold is automatically discharged to a secured hopper located within the gold room.

On a batch basis, the rough Knelson concentrates are transferred from the secured hopper to an Acacia intensive leach reactor (ILR). The concentrates are deslimed using water before caustic cyanide leach solution containing a proprietary "Leach Aid" is recirculated through the concentrate bed to rapidly dissolve the contained coarse gold. At the completion of the leach cycle, the concentrate residue solids are washed and discharged back to the grinding circuit. The high grade pregnant leach liquor containing the gold is then recirculated through an electrowinning cell, with the gold won onto stainless steel mesh cathodes. The high-quality won gold is then direct smelted to saleable bullion product.

The gravity recovery circuit at Union Reefs is shown diagrammatically in Figure 17-5. No changes to the existing gravity circuit are envisaged.

17.3.5 Flotation Circuit

Cyclone overflow is first screened to remove trash, then flows to the conditioning tank, where copper sulphate, frother and SIBX (xanthate collector) are added and given time to mix. Copper sulphate and SIBX can also be added upstream to the mill circuit if this is found to be beneficial.

Conditioned concentrate overflows to the rougher flotation cells. Rougher concentrate reports directly to the final concentrate thickener. Rougher tails flows to the scavenger cells.

Scavenger cells flotation concentrate is pumped back to the cleaner flotation cells, while scavenger tails is pumped to the tailings thickener.

Cleaner concentrate reports to the final concentrate thickener. Cleaner tailings are usually returned to the rougher cells, but may also be returned to the scavenger cells. The next stage of design will also enable tailings to be pumped to the CIL circuit to allow for low recovery Maud Creek transitional mineralization. Refer to Figure 17-6 for an isometric overview of the flotation area.

17.3.6 Concentrate Thickening Filtration Circuit

Flotation concentrate is first settled in a high rate thickener. The thickener underflow is further dewatered in a filter press as per conventional practice. It is dumped to ground and contained in a storage shed where it can be bagged, then loaded into sea containers and onto trucks. Filtrate and thickener overflow are returned to the process water tank. A separate process water system has been allowed for at this stage of study to eliminate concerns with reagent contamination between the CIL and the flotation circuits. Figure 17-7 provides an isometric overview of the concentrate filtration, concentrate storage and bagging area.

Figure 17-6: Proposed Flotation and Concentrate Thickening Isometric Drawing

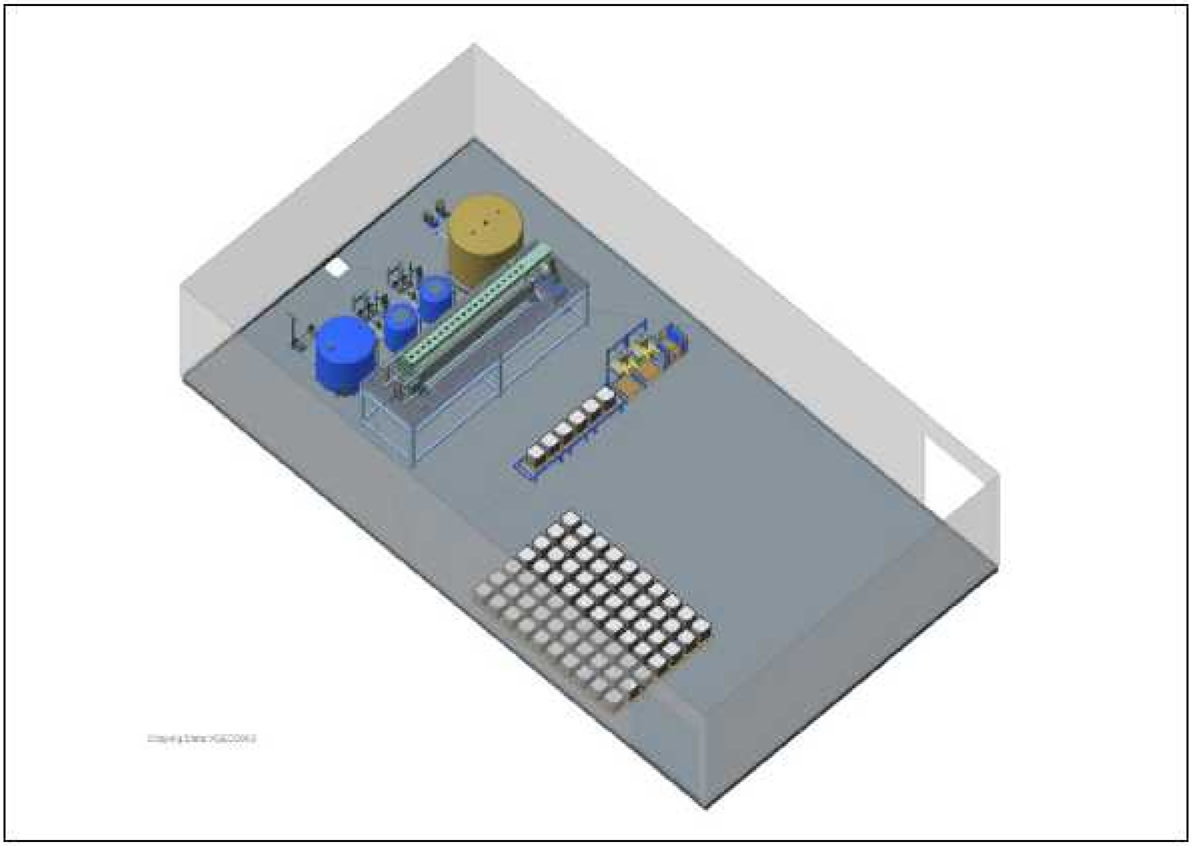


Figure 17-7: Concentrate Filter, Storage and Bagging Area Isometric Drawing

17.3.7 Tailings Circuit

Flotation tailings are thickened and the underflow pumped to the tailings storage facility. The overflow is returned to the process water tank. The next stage of design will also enable tailings to be pumped to the CIL circuit to allow for low recovery Maud Creek transitional mineralization.

17.4 Reagents and Services

17.4.1 Reagents

Xanthate, copper sulphate and Flocculant are delivered to site in solid bulk bags. They are mixed with water in batches, then pumped to where they are required.

Frother is delivered to site as a liquid, stored in a tank and pumped to the flotation area.

Grinding media is provided in 44 gallon drums or in bulk sea containers where it is unloaded into concrete bunkers.

Major reagent consumption expectation based on testwork is shown in Table 17-2.

Table 17-2: Reagent Consumption Rates

Reagent	Consumption Rate (g/t)	Consumption Rate (tpa)
Frother	20	10
Collector (SIBX)	125	82.5
Activator ($CuSO_4$)	50	16
Flocculant	70	16.6
SAG Mill Grinding Media	2810	1405

17.4.2 Water

The existing facilities at Union Reefs Processing Plant will be used to draw the required raw and potable water.

Process water consists of thickener overflows, tailings dam reclaim water, and brine from the water treatment plant, with make-up from the raw water. A separate process water system has been allowed for at this stage of study to eliminate concerns with reagent contamination between the CIL and the flotation circuits. Confirmation is required to whether it is required and the level of associated risk. It has been allowed for in the capital cost estimate.

Additional discussion on water is provided in Section 18.6.

17.4.3 Air

Plant air will be supplied by a compressor and air receiver package. Instrument air will be supplied by plant air ran through a drying and air receiver package. Low Pressure (LP) flotation air will be supplied by three air blowers running 2 duty / 1 standby. The full plant and LP air requirements has been allowed for in the capital costs although it is expected there will be some overlap and capital costs can be reduced at the next phase of study.

17.4.4 Power

Power is provided by an incoming line from the Northern Territory grid. Additional information on power supply and reticulation is provided in Section 18.3.

17.5 Flotation Concentrate

17.5.1 Flotation Concentrate Grade

Flotation concentrate composition from the pilot plant runs was reported as shown below in Table 17-3:

Table 17-3: Pilot Plant Flotation Concentrate Composition

Tests	Au g/t	S%	As%	Fe%
1996	47.5	17.2	3.1	20.8
1997	51.4	19.5	3.8	21.9
2006	40.7	22.2	3.9	-
Average	**46.5**	**19.6**	**3.6**	**21.3**

Comprehensive analysis of the concentrates was not undertaken for all pilot campaigns. Additional assays from the 1996 campaign are presented below (indicative only).

- Ag - 20 g/t
- Cu - 1600 ppm
- $C_{(total)}$ - 2.79%
- $C_{(organic)}$ - 0.04%
- SG - 3.2.

An XRD analysis on a combined concentrate from the same piloting campaign is shown in Table 17-4 providing approximate mineral proportions based on visual estimates.

Table 17-4: Pilot Plant Flotation Concentrate XRD Analysis

Mineral	**%**
Pyrite	40
Arsenopyrite	25
Chalcopyrite	3
Marcasite	3
Galena	<1
Sphalerite	<1
Goethite	1
Quartz	10
Muscovite /Sericite	10
Chlorite	2
Carbonate (dolomite)	6
Total	**100**

This is the best available indicator of what concentrate specification is likely to be produced from the flotation plant. It does not include gold recovered from the gravity section. A gravity concentrate grade of 45 g/t Au has been selected for design purposes and transportation costs of the concentrate. This is considered to be conservative. It is expected cleaning of the concentrate, flash flotation and/or a finer grind will improve the concentrate grade at similar recovery. Metallurgical testwork will be required to generate a flotation concentrate sample to provide to customers for their own testing.

Note that the first two pilot plants included periods with and without cleaner flotation, while the design includes a cleaner stage.

17.5.2 Concentrate Sale Options

The likely market for the gold concentrate is China. This has been selected as the base case. Within Australia, there are a few operations that could potentially take the concentrate, but transport costs are high and at first inspection, it has been assumed that the customers do not have sufficient capacity available to treat the full production tonnage. These options are discussed further in Section 17.7.

Newmarket Gold' s Fosterville Gold Mine in Victoria could potentially treat the concentrate through its BIOX® circuit. Biomin (the BIOX® technology providers) has indicated that this is technically feasible and extensive testwork and engineering study has been undertaken on this option. However there is insufficient capacity for all the concentrate, the transport cost is likely to be quite high, and recovery would be affected by the preg-robbing characteristics. The scope of the study specifically excludes this as a sale option at this time.

17.5.3 Concentrate Terms

The payable gold content depends on the tonnage sold as well as gold and impurity grades.

Under Chinese regulations, the gold concentration must be over 40 g/t to be classified as a concentrate. Most metal sulphide concentrates imported into China have arsenic importation limits of 0.5%, gold concentrates do not have this limitation. The arsenic grades in the Maud Creek gold concentrate are well above this level (approximately 3.6% based on piloting) as shown in Table 17-3. Lower gold grades are classed as ore and incur higher taxes and arsenic grade is likely to result in importation restrictions. Maud Creek concentrate should have no difficulty exceeding the 40 g/t threshold.

The major impurity to consider is arsenic. Chinese customers will accept material in the 3 – 5% range expected from Maud Creek, but it incurs a penalty based on the grade.

A number of indicative terms have been provided for the Maud Creek concentrate, the best of which would realise about 95% payable gold content, with the following deductions:

Treatment charge (TC) AUD10 / tonne concentrate

Up to 97% could be realised with higher gold grades of >60 g/t. The arsenic penalty is expected to be AUD4/t concentrate for every 0.1% arsenic over 0.2% arsenic in concentrate.

This is based on informal discussions with market players and experience on similar projects, and closely agrees with values used in the Core Process Engineering report 140-001 of September 2011. Further benchmarking has been undertaken. The next phase of study needs to progress concentrate sale and terms negotiations.

17.5.4 Transportation costs

This is discussed in Section 19.1 Concentrate Transport.

17.6 Process Processing Risks and Opportunities

A number of process risks and opportunities have been covered in detail throughput the report. The risks are largely mitigated due to the significant testwork that has been undertaken on the Maud Creek mineralization, the favourable metallurgical behaviour and the exiting Union Reefs Processing Plant available to process the Maud Creek mineralization. A simple conventional flowsheet is required. A summary of the key process risks and opportunities are given below.

17.6.1 Risks

- Metallurgical samples were biased to shallower samples.
- The mineralization is hard and abrasive, which may present a number of issues. These include high media consumption, increased milling power requirements, and mill liner wear, which will drive up operating costs.
- The gravity testwork gold recovery is highly variable, however overall gold recovery remains high.
- There is a lack of information on flotation testwork performed on the transitional mineralization. Although the transitional mineralization only accounts for around 11 percent of the total mill feed, it will still have an effect on the project value. It is recommended some further flotation testwork be done on the transitional mineralization.
- Development of feed and concentrate grade versus recovery relationships needs to be developed. This may require testing on some lower grade samples (testing has been biased to higher grades).
- With the change to Union Reefs as the base case processing option, additional review of the oxide mineralization behaviours is required although it makes up a relatively small proportion of the overall feed.
- There are a number of downstream processing options available. More work is required at the next phase of study to confirm that the direct sale of concentrate remains the most favourable. The risk of future curtailments of gold concentrate into China (possibly through tightening of the arsenic importation limits) warrants further consideration.

17.6.2 Opportunities

There has been limited metallurgical testwork done on flash flotation. The piloting that has been done showed it was effective but it has not yet been included in the flowsheet. There may be potential increase recovery and/or concentrate grade with a marginal capital cost increase. This remains a flowsheet opportunity and will be examined in more detail during future phases of work.

Finer grinding may improve plant performance. As with the other opportunities, this should be examined in more detail in the next phase of the study.

There may be an opportunity to increase concentrate grade but at a marginal reduction in gold recovery. Depending on the final downstream processing option selection, the recoveries associated with a higher concentrate grade may be justified by the reduction in concentrate transportation costs.

The use of the Union Reefs Processing Plant allows for potential increases in the throughput.

17.7 Toll Treatment Options for Product Concentrate

The Maud Creek deposit contains refractory gold mineralization, as demonstrated by testwork. This refractory nature is due to the close association between the gold and sulphide mineralization. Complicating things is the presence of carbonaceous mineralization presenting 'preg-robbing' behaviour during cyanide leaching, resulting in the mineralization being classified as 'double-refractory' .

Due to the gold being sulphide locked, it requires alternative treatment to direct conventional cyanide leaching. The relatively small production rate for Maud Creek concentrate and short LoM makes the construction of a dedicated refractory gold treatment plant for extraction of gold uneconomic. The best option is to produce a gold concentrate for treatment at a separate facility, which opens up the options to all cyanide pre-treatment technologies and other processing methods for recovering gold from refractory mineralization. Starting with the "Maud Creek Gold Project Processing Options Review" undertaken by Signet in 1998, each of the major pre-treatment technologies has been considered in the past. This latest assessment revisited these previous studies and undertook a preliminary review of the best options available to the Project, including potential 3rd party processing options in Australasia and abroad. Brief descriptions of the main options are provided below including the initial findings.

17.7.1 Ultrafine Grinding

Ultrafine grinding (UFG) involves the use of a stirred medium grinding mill to grind and liberate locked gold from sulphides by grinding as fine as 1 - 10 µm. The oxidised mineralization can then be treated by conventional cyanide leaching. There is generally an economic optimum point between cost and recovery resulting in a grind coarser than this. Ultrafine grinding was tested previously and found to be relatively ineffective. In the Signet study, median gold extraction rates of 61% at 100% -125µm only increased to 65% at 100% -20 µm. In a later report (by Hydromet Innovations for Mercator Gold, 2007), UFG to P_{80} of 15 µm increased extractions by between 6 and 22% of total gold, to values in the range 66 to 81%. In the 81% extraction case, gold extraction was increased to 86% by adding alkaline pre-treatment, but this would require a toll treatment operator to tailor their process to incorporate this pre-treatment. For this concentrate alkaline pre-treatment would likely be infeasible for the available benefit. Gold extraction of 71% was reported from UFG in a report by Independent Engineers (Interim Findings Fatal Flaw Review Maud Creek Project, 2005). Neither does it account for the preg-robbing of the mineralization. Collectively, these results make UFG unattractive based on the cost versus increased gold extraction and recovery.

17.7.2 Bio-oxidation

Bio-oxidation uses microbes to oxidise the gold containing sulphide matrix, breaking down the sulphide matrix surrounding the gold particles and exposing them for further treatment. The oxidised mineralization can then be treated by conventional cyanide leaching. The Signet tests were successful in achieving extraction rates up to 95%, although there were some challenges with regard to high carbonate and arsenic (As) levels, which would have to be resolved for effective implementation. The Hydromet Innovations study returned a gold extraction rate of only 72%, and 86% was reported by 'Independent Engineers' . The key barrier to pursuit of this option, however, is that of the three BIOX® plants identified in Australia, two (Beaconsfield and Wiluna) are in care and maintenance and are remote to the site; Tasmania and the northern WA Goldfields respectively. The one remaining operating site (Fosterville) has been assessed in the past. It does not have sufficient capacity for the entire production rate. Fosterville is owned by Newmarket Gold and they could bring considerable technical expertise to this option however a standalone BIOX® plant is not justified. Part off-take could be future opportunity but will need to consider the amount of preg-robbing material in the feed.

17.7.3 Pressure Oxidation

Pressure oxidation is a process where sulphur removal is carried out under high pressure and elevated temperature in an autoclave. The oxidised mineralization can then be treated by conventional cyanide leaching. The Hydromet Innovations study reported POX extraction of 98%. However, there are no POX autoclaves available for gold concentrate in Australia. Macraes (New Zealand) is an option and it is understood it is prepared to accept some concentrate, but the indicative treatment terms provided for other operations were marginally less favourable than those for the sale of concentrate to Chinese processors and has not been pursued further at this stage. It remains and option. It is a good option for arsenic containing concentrates as it produces a relatively stable precipitate.

17.7.4 Ultrafine Grinding followed by Moderate Pressure Leaching (Albion/Activox/CESL)

The Albion, Activox and CESL processes all involve oxidative leaching at atmospheric (Albion) or under moderate temperature and pressure conditions (Activox and CESL), with oxygen injection (Albion and Activox) or oxygen and HCl addition (CESL) at elevated temperatures (100^{o}C for Albion and Activox, 150^{o}C for CESL). The oxidised mineralization can then be treated by conventional cyanide leaching. A report by Independent Engineers (Interim Findings Fatal Flaw Review Maud Creek Project, 2005) indicated gold recoveries of 86% for Activox alone, and for the combined Albion / CESL processes. An Activox test in the Hydromet Innovations report achieved gold extraction of 92%. These processes are capital cost inhibitive due to the short life of mine and low tonnages.

17.7.5 GEOCOAT®

The Geocoat® process involves coating a carrier rock with concentrate slurry, then stacking the material onto an impervious pad and performing bio-oxidation. Independent Engineers reported gold extraction of 81%, and poor performance on a number of other grounds resulted in this being rated as the lowest preference in the range of ten options in their assessment.

17.7.6 Roasting

Roasting oxidises the sulphide concentrate releasing SO_2 gas. The oxidised mineralization can then be treated by conventional cyanide leaching. The KCGM owned Gidji roaster is shutting down and is not designed for arsenic in the concentrate, and is therefore not an option. The Kanowna Belle roaster can process arsenic contaminated concentrates, does currently have some available capacity but no formal approach has been made to ascertain how much. Transport costs (both in the vicinity of Kalgoorlie, WA, is an additional disincentive; therefore this option has been disregarded.

17.7.7 Selected Downstream Processing Option

Preliminary techno-economic assessment of the above options shows the terms offered by Chinese processors for direct smelting of the concentrate, which includes allowance for arsenic content, to be the most favourable option. This option is used by part of KCGM' s concentrate, Evolution Mining' s Mt Carlton Operation and KBL Mining' s Mineral Hill' s Pearse Gold concentrate. Further details of the preferred option are provided in Section 19. At the preliminary level of review, this option appears to offer the most favourable outcome, and is the primary one being pursued.

There may be an opportunity to use a mixture of the third party concentrate treatment options, due largely to available capacity, but this will make the downstream processing and sale process more complex. At this stage just the single sale option is being considered.

17.8 Stand-alone processing Plant Option

SRK completed a study to assess and define the requirements for the options of a Processing Plant on site at Maud Creek. The following is provided as background to the work undertaken to support the economic parameters presented in the comparison of the options. The Union Reefs Processing Plant option is the preferred case being presented in this PEA.

This option would require construction of a Tailings Storage Facility, increased water management, power and infrastructure requirements while providing paste fill for the proposed underground operations.

The current study reviewed the previous metallurgical testwork data and supporting engineering studies. From this body of work, a simple, conventional, inexpensive and low-risk flotation circuit has been selected, allowing flexibility with respect to circuit configuration and downstream processing. The base case presented is for a standalone mill and flotation concentrator plant at the Maud Creek site, producing a gold sulphide concentrate for sale or possible in-house processing. A gravity product will also be produced and either smelted on site or smelted and refined in Australia at one of the gold refiners.

The flowsheet consists of a single stage crushing and closed circuit SAG milling circuit, followed by centrifugal style gravity concentration, simple rougher / scavenger / scavenger cleaning flotation circuit followed by concentrate dewatering was chosen as the base case flowsheet. The inclusions of a pebble crusher and/ or a flash flotation would considered in a future work but have not been tested to the same extent as the flowsheet selected.

Downstream processing options for refractory gold mineralization and concentrates were considered at a preliminary level. The direct smelting of flotation concentrates option has been selected. This involves concentrate being dewatered, bagged, stored in shipping containers and transported by road, rail then ship to China. A plant capacity of 500 ktpa has been selected by SRK to support the mine production capability and as it strikes a balance allowing for a modular style of design and construction and management of the capital cost and reduce site construction.

17.8.1 Stand-alone Capital and Operating Cost Estimates

The stand-alone Processing Plant capital and operating cost estimates are summarised in Table 17-5 and Table 17-6.

Table 17-5: Stand-alone Processing Plant – Capital Cost Estimate

Item	Total (AUD M)
Process Plant	58.7
Surface Infrastructure	5.2
TSF Construction	12.0
Paste Fill Plant	4.0
Sustaining Plant	10.9

Table 17-6: Stand-alone Processing Plant – Operating Cost Estimate

	AUD/t
Processing	34.94
Concentrate transport	17.11
Site G&A, Indirects	13.00

18 Project Infrastructure

18.1 Site Access

A suitable access road connecting the Maud Creek project site to the Stuart Highway is required to facilitate access. The options are illustrated in Figure 18-1.

The existing unsealed access road to the Maud Creek site into Katherine, heading south from the mine was used during previous mine operations to haul material from site. The access road is linked to the Stuart Highway by Ross Road, a sealed public road which is regularly used by horticultural landholders in the area. The access road crossed Gold Creek just south of the existing mine pit and during the wet season, the road is often impassable and experiences significant damage.

The existing access track would need to be upgraded and have a crossing installed over Gold Creek to enable transport operations to continue during the wet season both into and from site. This route has been approved by the Northern Territory Government as a Right of Way and General Service easement. A second approved Right of Way and General Service easement runs north from the Mine to Gorge Road. This provides the site with a second access point to Katherine.

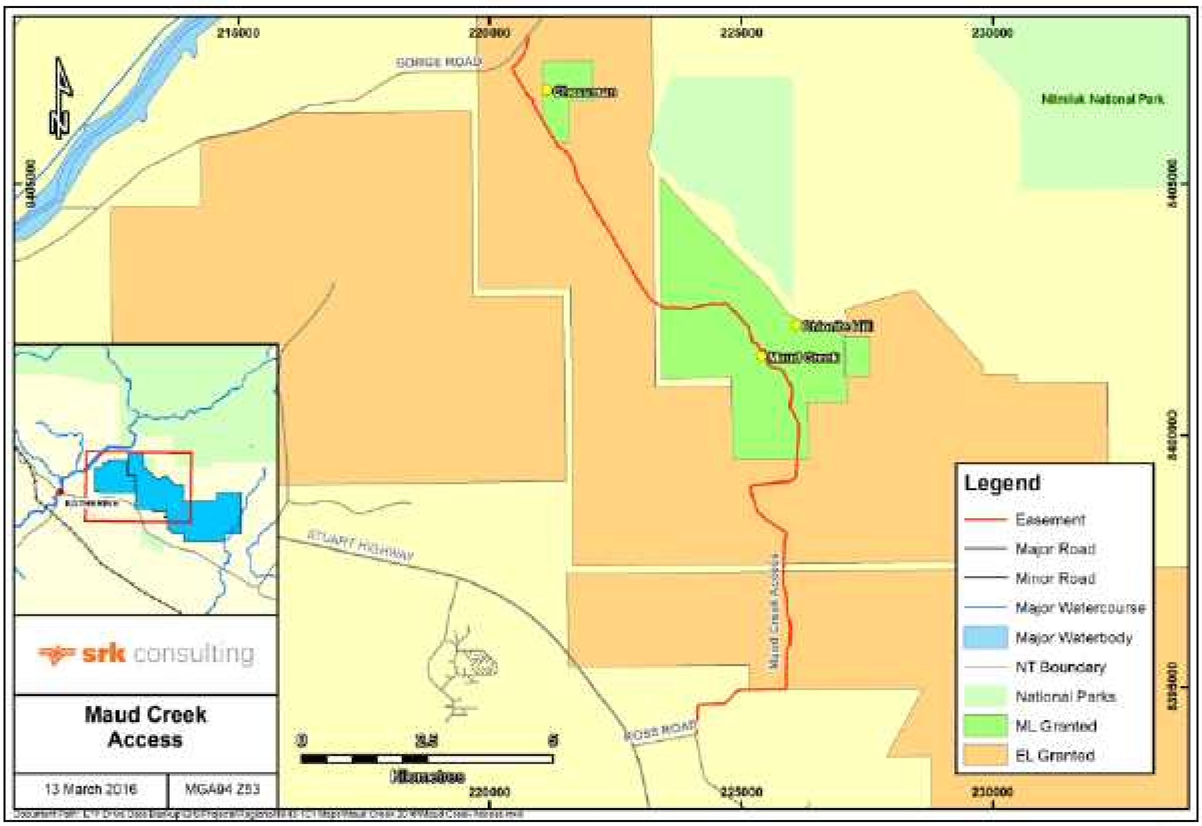


Figure 18-1: Site Access Roads

18.2 Surface Infrastructure

The Maud Creek Gold Project surface facilities are representative of a modern and conventional flotation style concentrators and underground mining operation. The site comprises the following:

- Office and administration complex, including change house;
- Store and laydown facilities;
- Heavy underground equipment workshop;

- Temporary surface mineralization stockpiles and waste stockpile area;
- Maud Creek Open Pit Mine and portal;
- Ventilation exhaust raises;
- Ventilation intake raise; and
- Raw water storage to manage rainfall runoff.

18.3 Power Reticulation and Supply

Power demand estimates are based on the 0.5 Mtpa standalone processing case and are summarised in Table 18-1. Power requirements for the Processing Plant was estimated based on the process plant motor list (preliminary) and assumed power draw and utilisation. This included an allowance for administration buildings, workshops and lighting. This is now largely redundant as processing will be at the existing Union Reefs Processing Plant that already incorporates this. The overall Maud Creek demand is significantly reduced as a result. It impacts the selection of the preferred power source.

At the Maud Creek mine site, an allowance of 2 MW of power has been made for the underground mining operations for surface ventilation fans, UG pumping, compressed air and mining equipment.

Table 18-1: Power Demand

Description	Units	Value
Process plant	MW	2.9
Underground	MW	2.0

Three Maud Creek mine site power supply options were considered (all at a preliminary level only):

- Mains power
- Site generated power (diesel)
- Site generated power (gas).

It was assumed that whichever option was selected as the preferred base case, it would be provided by a third party.

The town of Katherine is linked by a 132 kV transmission line to the Darwin-Katherine Interconnected System which supplies regulated electricity to the region. The transmission and distribution of power in the Northern Territory is the responsibility of the government owned Power and Water Corporation (PWC).

The first option is for electrical power provided via a dedicated 22 kV overhead (O/H) transmission line from the Katherine Power Station (gas fired) to a substation adjacent to the Maud Creek mine site. A preliminary route and capital contribution estimates for the proposed 22 kV transmission line have been provided to SRK by PWC based on a 2007 study. The proposed layout is shown in Figure 18-2. While the direct line distance from Katherine to Maud Creek is ~ 20 km, the distance of this route is approximately 40.5 km. This is as a result of the Power Station being located on the west side of Katherine and a new powerline corridor being run to the south of the town, following the site access road. The route is shown in Figure 18-1. The previous PWC study route was approximately 32 km long, but the currently available easement is now further south that the proposed 2007 route. Much of the infrastructure already exists along the proposed line, which should improve constructability and the approval requirements.

The estimated capital cost provided for the construction of the transmission line is AUD5.7M. In accordance with PWC' s power networks policy, the capital contribution would be approximately AUD3.6M and a prudential requirement (bank guarantee) of AUD2.1M. In a standalone mine and processing facility, this option becomes attractive but with the base case assuming processing at Union Reefs, the capital cost requirement of this option is high for the 2.0MW requirement of the mine only and is not considered in further detail.

On site power generation using gas fired generator sets was investigated. The budget costs to install a 2” polyurethane/carbon fibre gas pipe from the Stuart Highway to site was estimated by PWC as AUD5-6.0M, which is higher than the capital contribution associated with the transmission lines and for the same reason is not considered in further detail.

The preferred option is to have diesel generator sets provided, owned and operated by an independent power provider (IPP), to supply the electrical power required by the mine. This is a low capital cost option that suits the power demand and relatively short LoM. The overall charge for power generation at Maud Creek is AUD0.2875/kWh.

The power supply at the Union Reefs Processing Plant has an existing connection to the Northern Territory power grid. Network and electricity supply tariffs have been provided at an average unit power cost of AUD0.2177/kWh.

A more detailed assessment of onsite power generation will be undertaken in the next phase of the pre-feasibility study.

During periods of power outages, a ‘black start’ diesel generating unit will be used to provide sufficient power for emergency lighting, mine egress and to operate other critical equipment drives. Back up diesel power for critical drives is already available at Union Reefs.

The 11 kV supply will be fed underground via a service hole to a substation that steps the voltage down to 1000 volts and the power is reticulated to the working areas via cable and distribution boxes.

On the surface, power will be distributed to substations throughout the plant and stepped down to 415 V for use by mechanical equipment components. Distribution between the substations and the processing facilities will be undertaken in pipe racks using cable trays.

At the next level of design, a more detailed scheme for the distribution of power will be developed to obtain market pricing on substations, switch rooms and other electrical infrastructure.

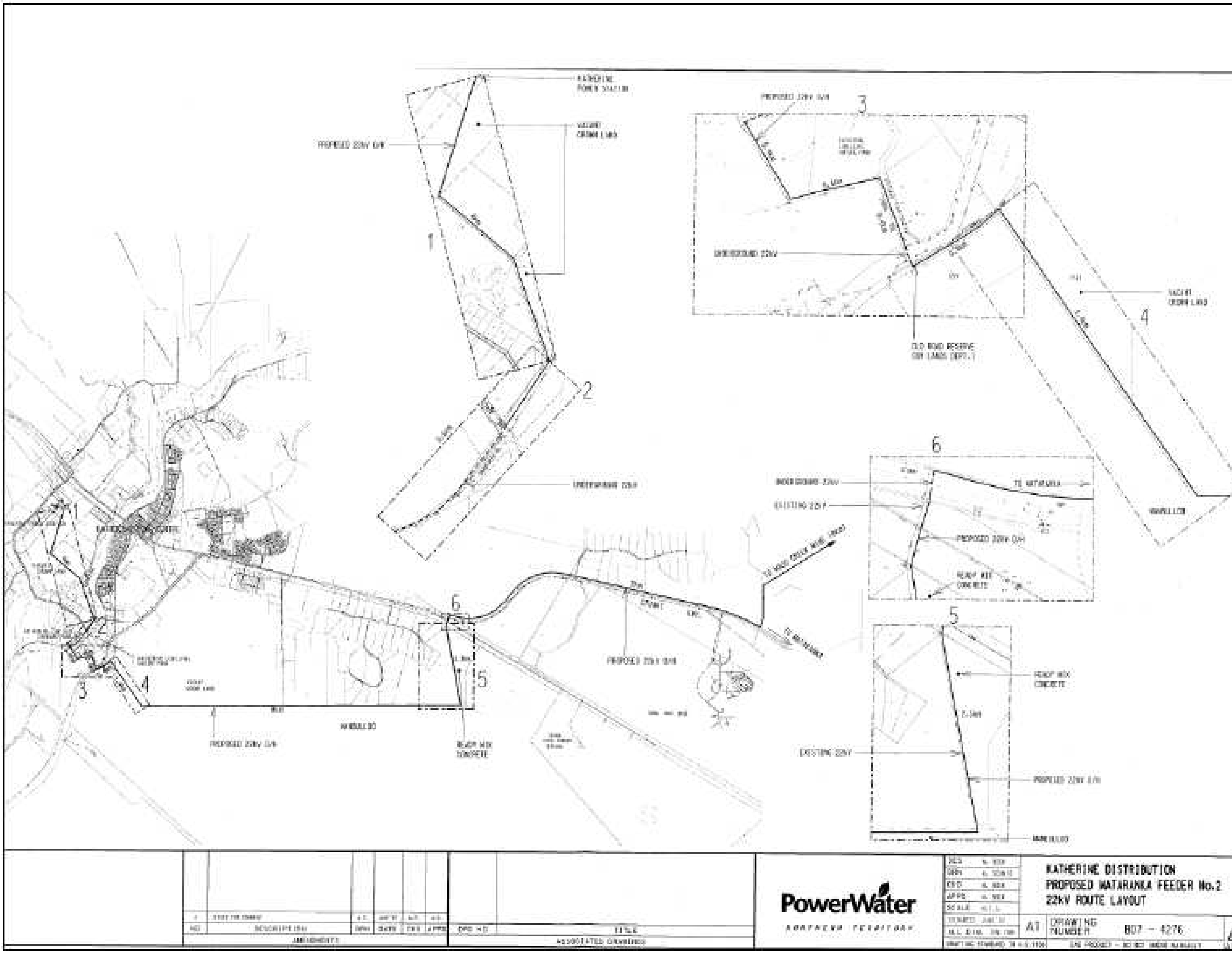


Figure 18-2: Proposed Power distribution

18.4 Water Supply

The raw water supply sources for the site include the:

- Groundwater seepage into the mine pit; and
- Potential runoff over catchments associated with the mine pit, waste rock dump, tailings facility infrastructure area, and sedimentation ponds.

The total catchment area reporting to the Maud Creek site of 25.74 km^2 was divided into three major sub-catchments, as shown in Figure 18-12. The areas of each sub-catchment are outlined in Table 18-2.

Water management of the site will need to be incorporated as a key factor in any future plans, to keep clean water clean and direct the contact water to appropriate containment system.

Table 18-2: Areal extent of catchment areas

ID	Area (km^2)
1	0.616
2	1.337
3	23.786

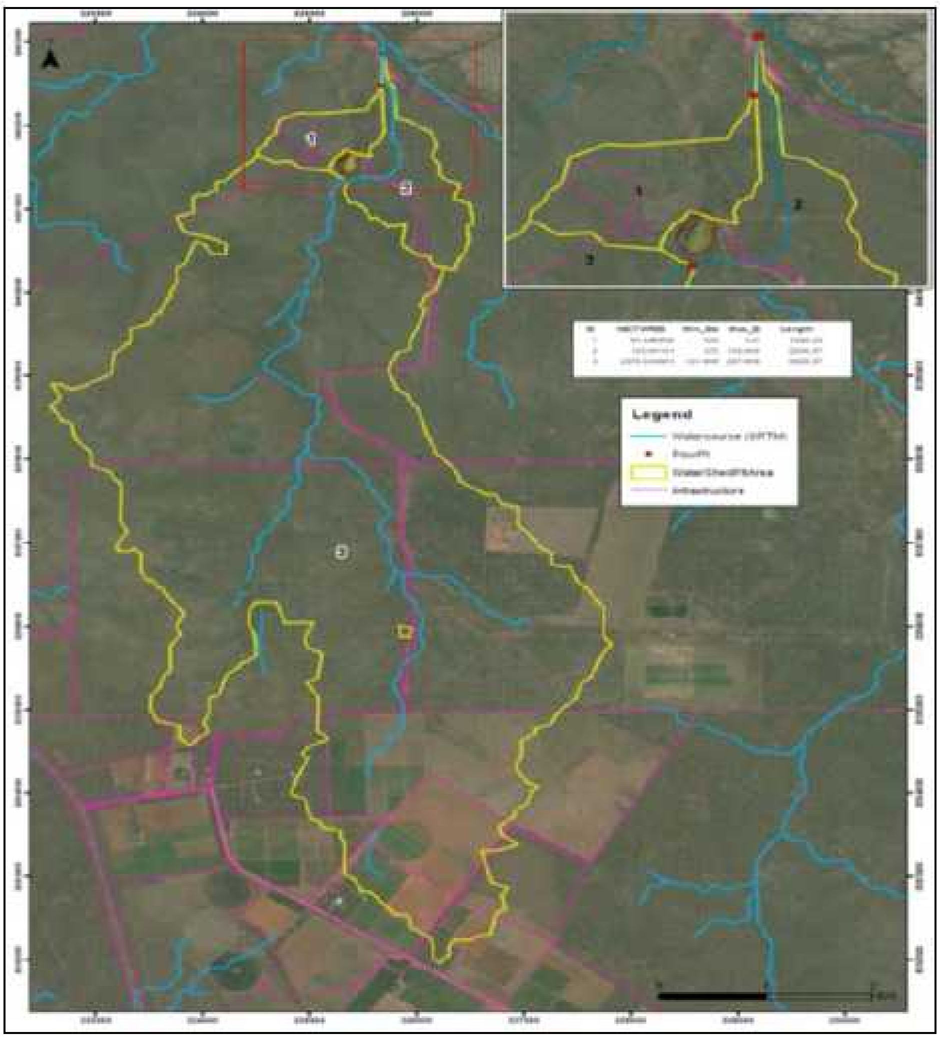


Figure 18-3: Catchment areas in Maud Creek site

18.5 Tailings Storage

The proposed processing route via the Processing Plant at Union Reefs, as such a Tailing Storage facility is not be required as part of the infrastructure requirements at Maud Creek.

The tailings disposal strategy for the Union Reefs process plant is to pump the tailings slurry to the Crosscourse pit tailings facility, with process waster recycled back to the Union Reefs Plant.

18.6 Surface Water Management - Hydrology

The primary objective for the surface water management of the Maud Creek project is to keep clean water clean and direct the contact water to appropriate containment systems. A number of water management strategies have been proposed, mainly utilising diversion and collection structures. The below-ground (Hydrogeology) aspects are discussed in Section 16.5.

The surface water management system for the Maud Creek project consists of the following elements:

- Flood protection bund to the east of the pit;
- Gold Creek diversion channel to the east of the pit;
- Channel to collect the contact water running off the flanks of the waste rock dump (WRD) which will report to the water pond;
- Water pond, receiving water from pit dewatering and surface runoff from WRD to feed the water treatment plant or to evaporate; and
- Emergency spillway to discharge contact water from the water pond into the environment in case of emergency.

The configuration of the water management system is shown in Figure 18-3.

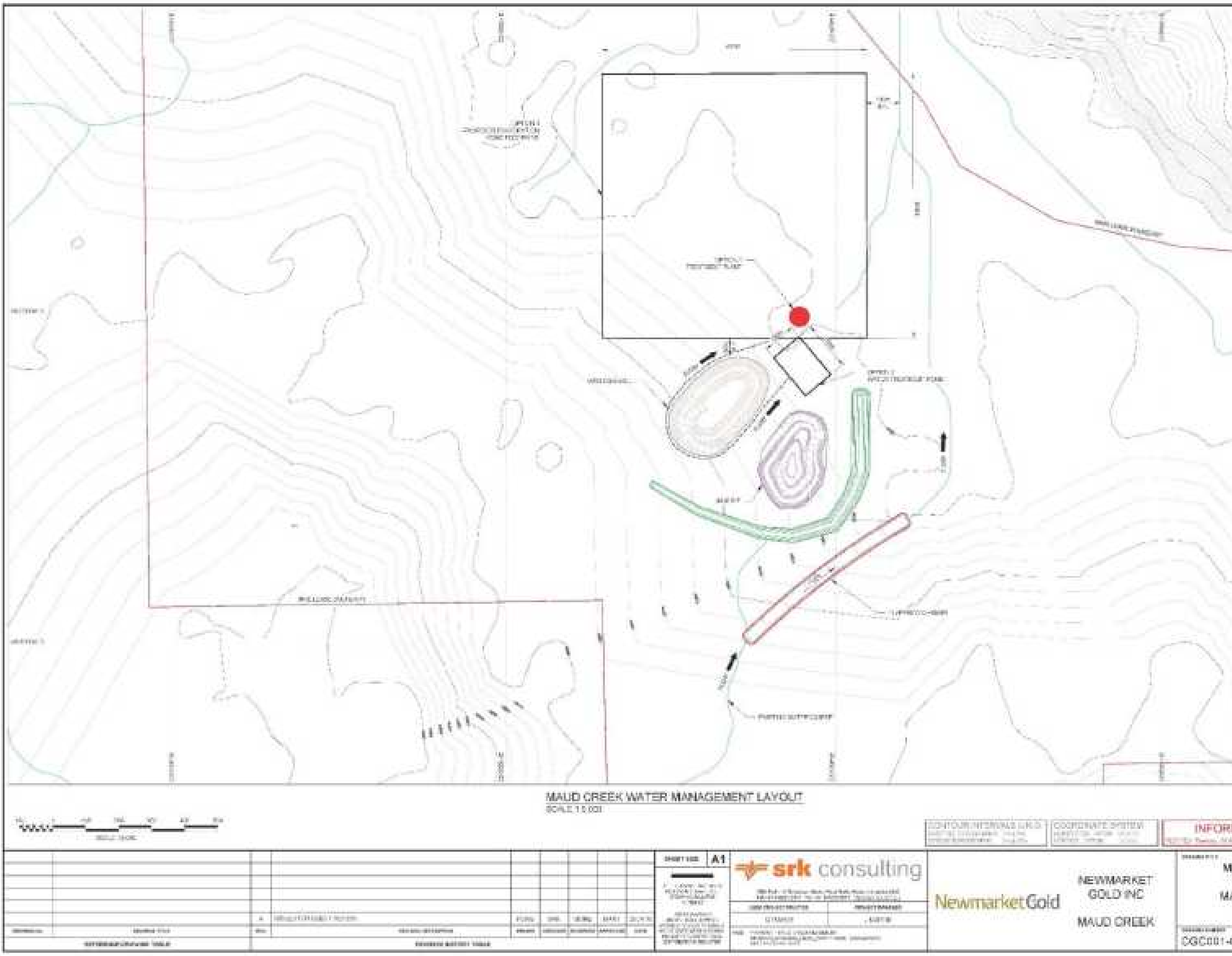


Figure 18-4: Water management configuration

18.6.1 Peak flow calculation

Peak flows represent the highest possible flow that a given catchment, channel or other feature can experience for a given storm/ precipitation event. The peak flows for the channels were calculated using the Australian Rainfall and Runoff (AAR): A Guide to Flood Estimation. The relevant design criteria are shown in Table 18-3.

Table 18-3: Peak flow calculation method extracted from ARR

Parameter	Value
Design Average Recurrence Interval (ARI)	1 in 100 years
Region Definition	Northern Territory
Method	Rational Method
Concentration time (tc) estimate	$t_c = 58\,L/\,(A10^{0.1}Se^{0.20})$
Runoff Coefficient (C2) estimate	variable with terrain slope
Peak Flow (QY)	$Q_Y = 0.278C_2\left(\frac{C_Y}{C_2}\right)I_{t_c}A$

Catchment characteristics are used to estimate concentration times and the runoff coefficient, C_2.

The intensity is determined by interpolating the Intensity, Duration and Frequency (IDF) data in Table 18-4 for the storm duration equal to the catchment' s concentration time.

Table 18-4: Intensities (mm/h) for various durations and return periods for Maud Creek

Duration				Average Storm Recurrence Interval (Years)						
hours	min	1	2	5	10	20	50	100	500*	1000*
0.08	5	110.0	141.0	178.0	200.0	230.0	271.0	303.0	365.8	394.4
0.10	6	103.0	132.0	166.0	187.0	215.0	253.0	283.0	341.5	368.2
0.17	10	85.6	110.0	138.0	155.0	178.0	209.0	233.0	281.4	303.2
0.33	20	65.8	84.1	105.0	117.0	134.0	157.0	174.0	209.9	225.9
0.50	30	54.7	69.9	86.6	96.7	111.0	129.0	144.0	173.1	186.2
1.00	60	37.0	47.2	58.5	65.2	74.6	87.0	96.7	116.3	125.1
2	120	22.8	29.1	36.1	40.3	46.2	53.9	60.0	72.2	77.7
3	180	16.6	21.3	26.5	29.6	33.9	39.7	44.1	53.2	57.2
6	360	9.6	12.2	15.3	17.2	19.7	23.2	25.8	31.1	33.6
12	720	5.6	7.2	9.2	10.3	11.9	14.1	15.8	19.1	20.6
24	1440	3.5	4.5	5.8	6.7	7.7	9.2	10.4	12.6	13.7
48	2880	2.2	2.9	3.8	4.4	5.2	6.3	7.1	8.7	9.4
72	4320	1.6	2.1	2.8	3.3	3.9	4.7	5.4	6.6	7.2

*Extrapolated values

In accordance with recommendations from AAR, peak flows were calculated using the Rational Method. This method is based on a simplified representation of the law of conservation of mass and the hypothesis that the flow rate in a catchment is directly proportional to the size of the contributing area and the rainfall intensity, with the latter a function of the return period.

The peak flows have been calculated for ARIs varying from 1 to 1000 years.

18.6.2 Gold Creek diversion bund and channel

A flood assessment has been undertaken to determine sizing requirements for a diversion bund and channel to divert storm flows around the pit. The two-dimensional flood routing software package FLO-2D was selected as the preferred software for the assessment because of its ability to simulate unconfined overland flow.

Model topography was sourced from the Shuttle Radar Topography Mission (SRTM) at a grid spacing of 30 m. The remaining model inputs are summarised in Table 18-5.

Table 18-5: Model inputs

Parameter	Value	Source / Comment
Design flood event	1-in-1000 year ARI	Assumed
Peak flow (m^3/s)	679.2	Calculated based on ARR recommendations
Time of concentration (hrs)	4.6	Calculated based on ARR recommendations
Manning' s n - watercourses and overbanks	0.06	Average values sourced from Hardcastle & Richards flood study 1998
Manning' s n - diversion channel	0.04	
Average diversion channel depth (m)	4	Hardcastle & Richards flood study 1998

The model was run at a grid spacing of 30 m to be consistent with the level of accuracy of the topographic information. The results are shown in Figure 18-4 and indicate that a minimum 9 m high bund is required to prevent flows associated with the 1-in-1000 year ARI event from encroaching the pit.

The diversion channel will have 4 m depth, 30 m wide and follow the natural slope. It will collect water from the catchment upstream of the pit. It will run south east of the pit and then discharge into the natural catchment.

A channel network was conceptually located to collect the contact water running off the flanks of the waste rock dump (WRD) which will finally report to the water pond.

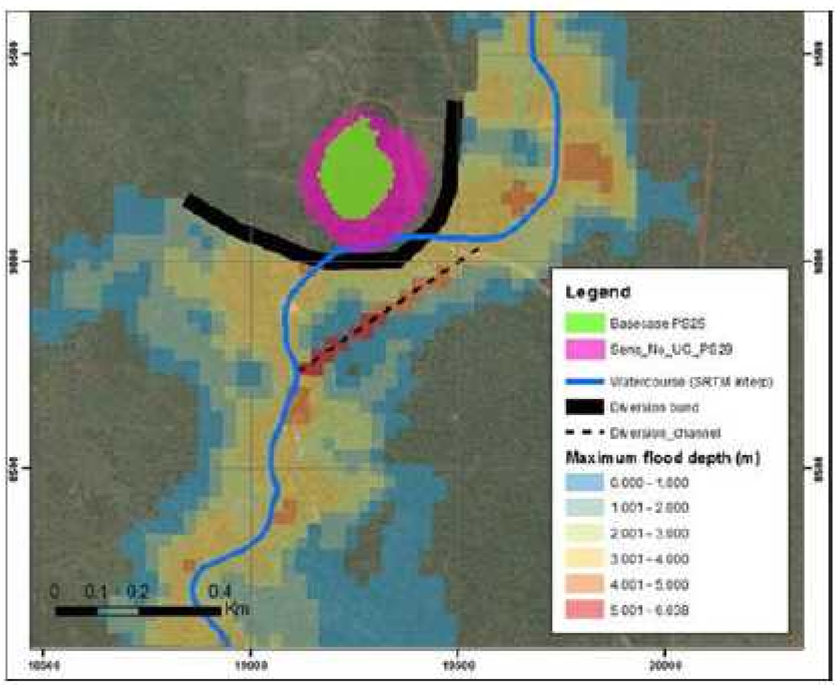


Figure 18-5: Flood routing results

18.6.3 Water pond

Two potential water pond sizes have been evaluated for the site as follows:

Option 1 considers an evaporation pond, large enough to contain and evaporate the water coming from the pit and also the runoff from the WRD.

Option 2 considers the installation of a water treatment plant with capacity of 3,200 m^3/d and a water pond to contain the excess water. Water from the pit and also runoff from the WRD will be directed to the water pond and from there send to the treatment plant.

Characteristics of the two water pond options are shown in Table 18-6 and their locations are shown in SRK Drawing CGC001-020. A spillway designed for closure will be constructed in the water pond. It will be a monitoring point for water quality and quantity.

The pond area required to rely on evaporation (Option 1) is almost forty-three times larger than considering a treatment plant (Option 2). It is recommended to explore both options when more details regarding water quality are available.

Table 18-6: Water ponds capacities

Water pond	Treatment Capacity (m^3/d)	Pond Area (m^2)	Pond Depth (m)	Pond Volume (m^3)	Freeboard (m)
Option 1	0	640,000	4.0	2,560,000	1.06
Option 2	3,200	15,000	2.5	37,500	0.68

18.6.4 Water demand and supply assessment

Two water balance models were developed to assess the mine water availability, supply and containment based on the water management system described in Section 18.8.

A monthly water balance was developed considering an evaporation pond (water pond Option 1) and a daily water balance was developed to consider the effect of the water treatment while evaluating a smaller pond. The models simulate precipitation, evaporation, pit-dewatering and water storages, for the two water pond options during the mine life (10 years). The raw water supply sources for the site include the following:

Groundwater seepage into the open pit; and

Direct precipitation over the water pond; and

Potential runoff over catchments associated with the open pit and waste rock dump.

The only water demand considered in the system is dust suppression at an estimated rate of 240m^3/d.

The daily model takes into account the ability to release water from the water treatment plant to the natural streams after checking water quality standards are met. Key findings of the monthly water balance for Option 1 are:

The water demand is met all the time during operations, Figure 18-5, where total inflows are larger than total outflows;

There is excess water in the system to be stored and accumulated while evaporation occurs;

The maximum accumulated water volume over the ten years of mine life is approximately 2Mm^3, Figure 18-6;

The total inflows are greatly influenced by the pit dewatering which decreases yearly; and

Rainfall and runoff is not a significant or reliable source of water for mine operations. This is due to the climatic characteristics of the site, which has episodic rainfall and high evaporation.

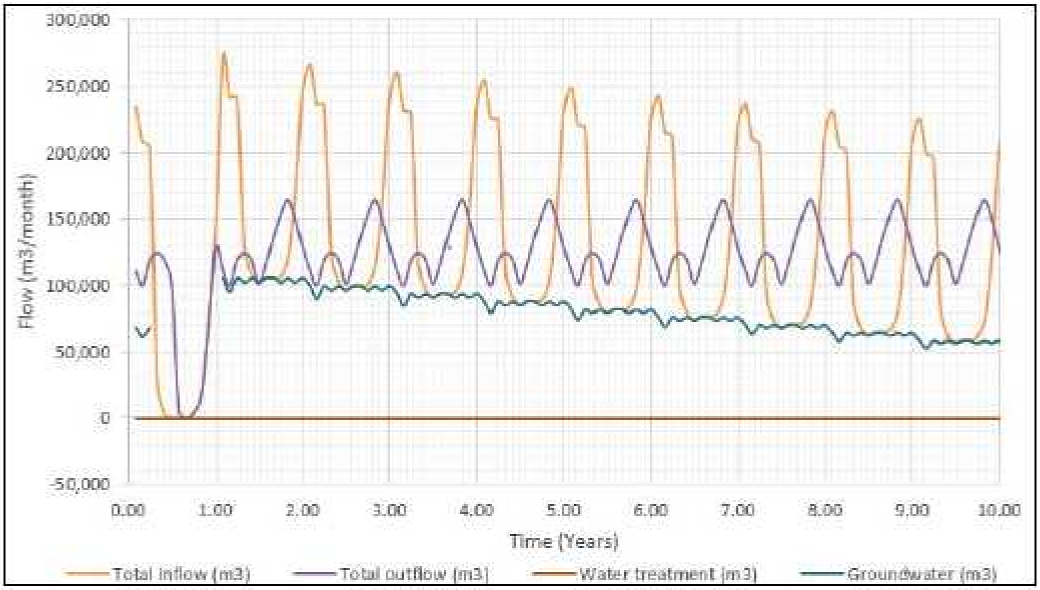


Figure 18-6: Total monthly inflows and outflows - Option 1 system

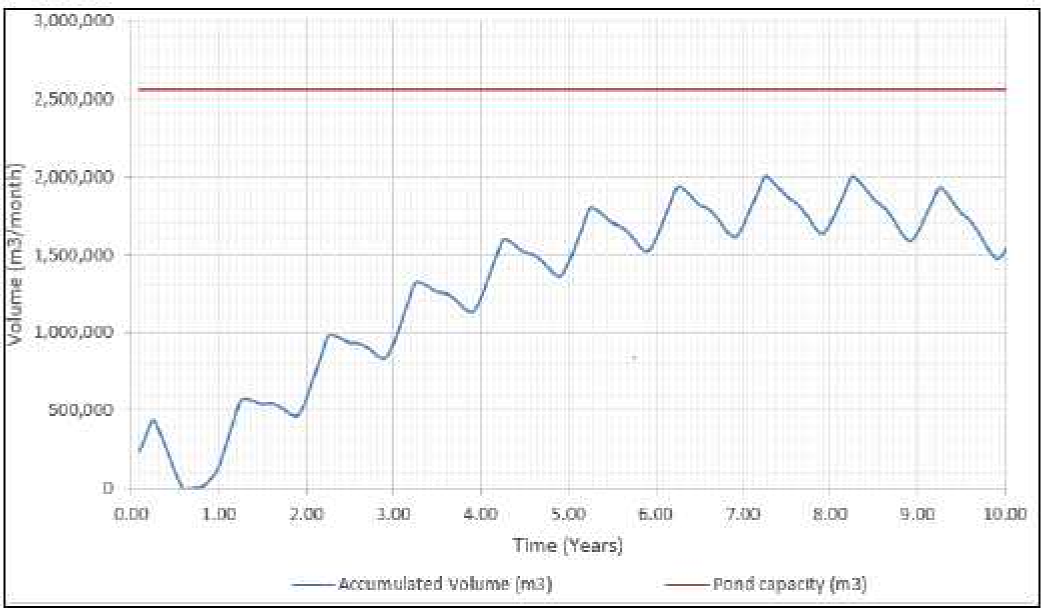


Figure 18-7: Monthly water accumulated volumes and pond capacity - Option 1

Key findings of the daily water balance for Option 2 are:

- The water demand is met all the time during operations, Figure 18-7 where total inflows are larger than dust suppresion requirement.
- The total inflows are greatly influenced by the pit dewatering which decreases yearly.
- Total inflows exceed total outflows during year two and three only.
- There is excess water in the system to be stored particularly during years two and three.
- The maximum accumulated water volume after treatment over the ten years of mine life is approximately 27,300m^3, Figure 18-8.
- Rainfall and runoff is not a significant or reliable source of water for mine operations. This is due to the climatic characteristics of the site, which has episodic rainfall and high evaporation.

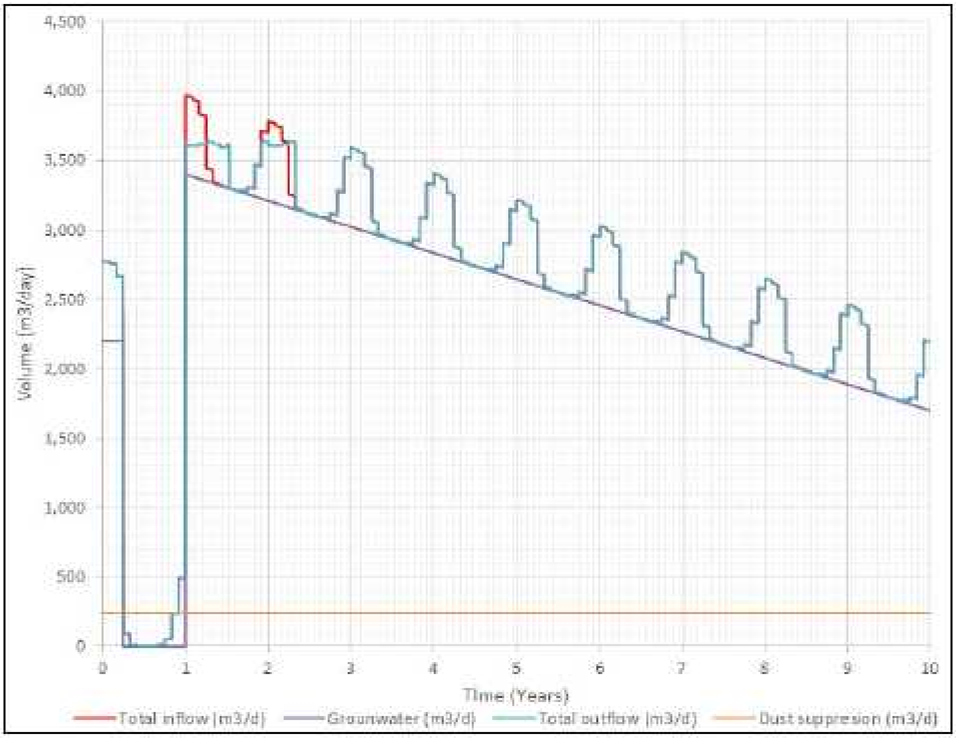


Figure 18-8: Total inflows and outflows - Option 2 system

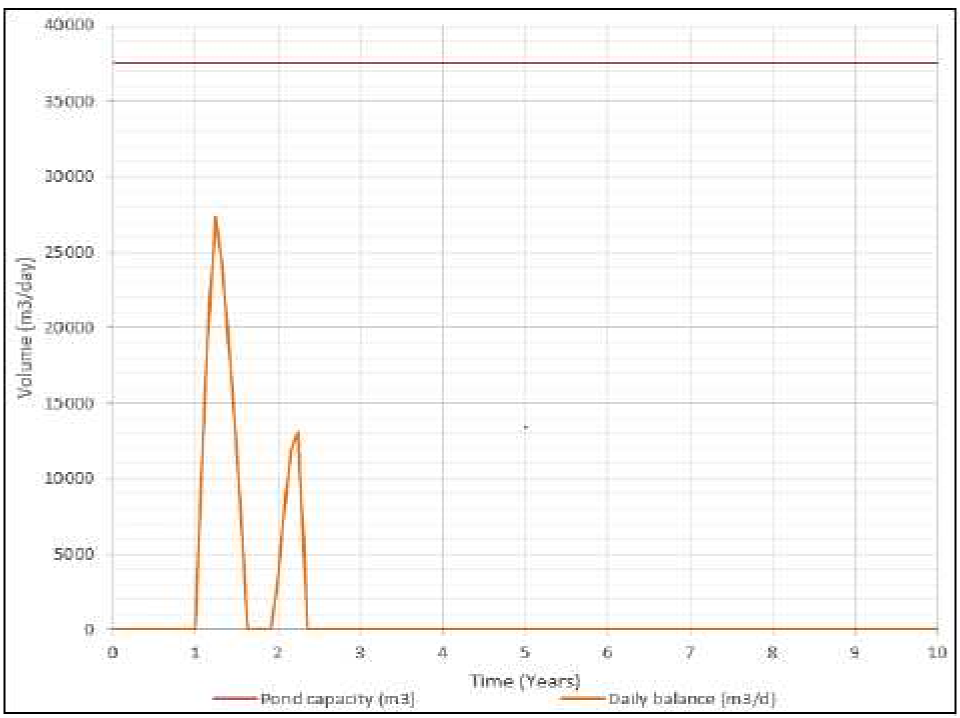


Figure 18-9: Water accumulated volume and pond capacity - Option 2

18.6.5 Preliminary cost estimate

A cost estimate have been prepared based on the preliminary schedule of quantities developed for major cost items for both Option 1 (Table 18-7) and Option 2 (Table 18-8). Average unit costs were taken from recent tenders received from WA contractors for construction work.

SRK has assumed a cost for a water treatment plant to be AUD5M.

Table 18-7: Preliminary cost estimate for surface water Option 1

ITEM	DESCRIPTION	UNIT	QUANTITY	RATE (AUD)	COST (AUD)
A	**Preliminary and General**				**5,897,374**
A.1	Mobilization, time related running costs, and de-mobilization; assumed 25% of the sum of all other costs	Lump Sum	1	5,897,374	5,897,374
B	**Earthworks**				**17,141,439**
B.1	**Clear / Grubbing / Topsoil Stripping**				
B.1.1	Clear, grub and rip pond footprint (assumed 150mm depth)	m^2	640,000	1	422,400
B.1.2	Strip topsoil from pond footprint (about 250mm depth)	m^3	160,000	4	641,600
B.1.3	Clear, grub and rip channel footprint (assumed 150mm depth)	m^2	14,625	1	9,653
B.1.4	Strip topsoil from channel footprint (about 250mm depth)	m^3	10,000	4	40,100
B.1.5	Clear, grub and rip Gold Creek diversion footprint (assumed 150mm depth)	m^2	28,150	1	18,579
B.1.6	Strip topsoil from Gold creek diversion footprint (about 250mm depth)	m^3	10,000	4	40,100
B.1.7	Clear, grub and rip bund protection footprint (assumed 150mm depth)	m^2	41,575	1	27,440
B.1.8	Strip topsoil from bund protection footprint (about 250mm depth)	m^3	20,000	4	80,200
B.2	**Excavation**				
B.2.1*	Excavate in situ material (pond) and either direct place into the diversion embankment or stockpile for re-use.	m^3	3,072,000	5	14,784,000
B.2.2*	Excavate in situ material (channel) and either direct place into the diversion embankment or stockpile for re-use.	m^3	14,040	5	67,568
B.2.3*	Excavate in situ material (diversion) and either direct place into the diversion embankment or stockpile for re-use.	m^3	110,000	5	529,375
B.3	**Compacted Earthworks**				
B.3.1	Spread, moisture condition, and compact to specification, excavated in situ fill material to form bund embankment	m^3	172,350	3	480,426
C	**Lining**				**6,448,056**
C.1	Supply and install HDPE liner to prepared base of pond	m^2	640,000	10	6,304,000
C.2	Supply and install HDPE liner to prepared base of channel	m^2	14,625	10	144,056
D	**Treatment plant**				
D.1	Supply and install treatment plant	Lump Sum	0		NA
TOTAL					**29,486,869**

*Includes 20% bulking factor

Table 18-8: Preliminary cost estimate for surface water Option 2

ITEM	DESCRIPTION	UNIT	QUANTITY	RATE ( )	COST ( )
A	**Preliminary and General**				**462,952**
A.1	Mobilization, time related running costs, and de-mobilization Assumed 25% of the sum of all other costs	Lump Sum	1	462,952	462,952
B	**Earthworks**				**1,560,002**
B.1	**Clear / Grubbing / Topsoil Stripping**				
B.1.1	Clear, grub and rip pond footprint (assumed 150mm depth)	m²	15,000	1	9,900
B.1.2	Strip topsoil from pond footprint (about 250mm depth)	m³	10,000	4	40,100
B.1.3	Clear, grub and rip channel footprint (assumed 150mm depth)	m²	14,625	1	9,653
B.1.4	Strip topsoil from channel footprint (about 250mm depth)	m³	10,000	4	40,100
B.1.5	Clear, grub and rip Gold Creek diversion footprint (assumed 150mm depth)	m²	28,150	1	18,579
B.1.6	Strip topsoil from Gold creek diversion footprint (about 250mm depth)	m³	10,000	4	40,100
B.1.7	Clear, grub and rip bund protection footprint (assumed 150mm depth)	m²	41,575	1	27,440
B.1.8	Strip topsoil from bund protection footprint (about 250mm depth)	m³	20,000	4	80,200
B.2	**Excavation**				
B.2.1*	Excavate in situ material (pond) and either direct place into the diversion embankment or stockpile for re-use.	m³	45,000	5	216,563
B.2.2*	Excavate in situ material (channel) and either direct place into the diversion embankment or stockpile for re-use.	m³	14,040	5	67,568
B.2.3*	Excavate in situ material (diversion) and either direct place into the diversion embankment or stockpile for re-use.	m³	110,000	5	529,375
B.3	**Compacted Earthworks**				
B.3.1	Spread, moisture condition, and compact to specification, excavated in situ fill material to form bund embankment	m³	172,350	3	480,426
C	**Lining**				**291,806**
C.1	Supply and install HDPE liner to prepared base of pond	m²	15,000	10	147,750
C.2	Supply and install HDPE liner to prepared base of channel	m²	14,625	10	144,056
D	**Treatment plant**				
D.1	Supply and install treatment plant	Lump Sum	1	5,000,000	5,000,000
TOTAL					**7,314,760**

*Includes 20% bulking factor

18.6.6 Conclusions

Water management of the site has been incorporated as a key factor, to keep clean water clean and direct the contact water to appropriate containment systems. A preliminary analysis has been completed to prevent flows associated with the 1-in-1000 year ARI event from encroaching the open pit.

The designed infrastructure includes a flood protection bund to the east of the pit, a diversion channel to maintain Gold Creek to the east of the pit has been conceptually designed and a dedicated water pond for the excess water.

Based on the available data, the preliminary water balance indicates the water demand is met all the time during operations. There is excess water in the system to be treated and managed on-site.

18.7 Surface Haulage

A suitable access road connecting the Maud Creek mine site to the Stuart Highway is required to facilitate the delivery of ROM material to the Union Reefs Processing Plant. Three route options are illustrated below in Figure 18-9.

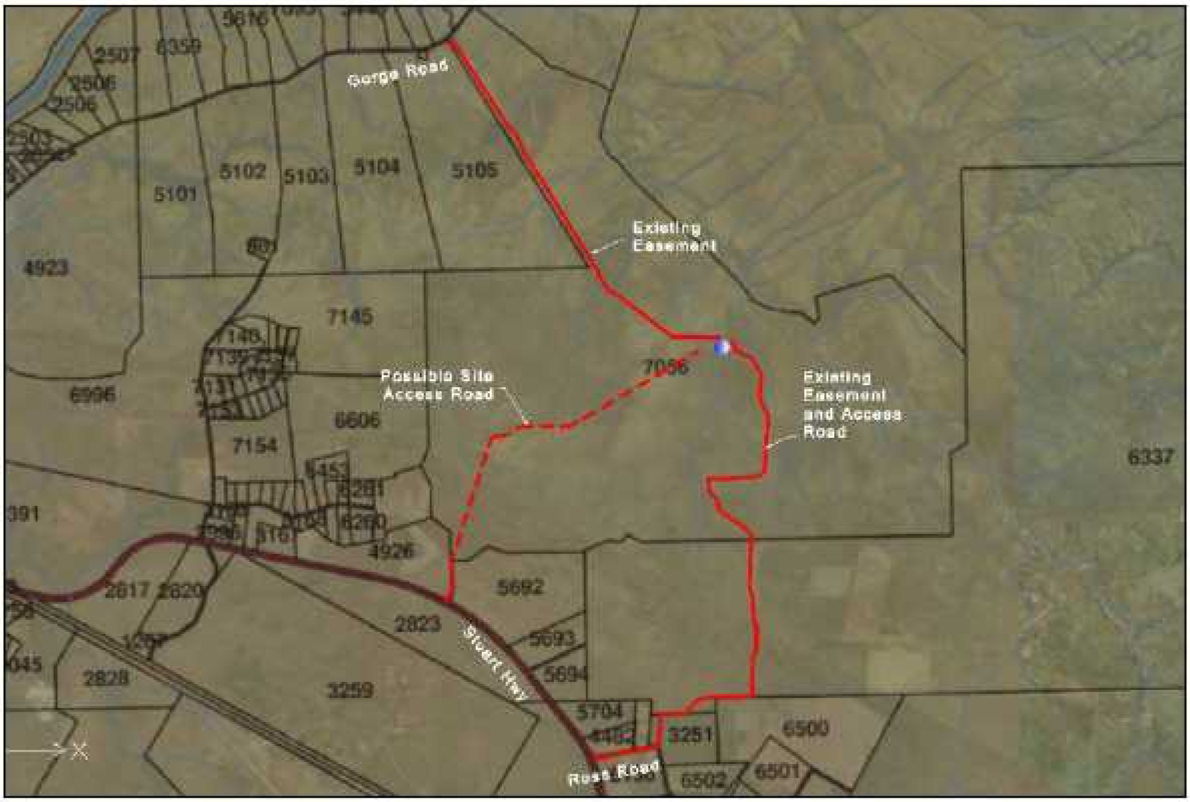


Figure 18-10: Maud Creek Site Access Roads

The use of rail via the Katherine Freight Terminal was not considered at this phase of study because of the relatively short haulage distance, the additional rehandle costs, the capital cost of a new siding and laydown area at Union Reef, the required approvals and access issues and the short LoM.

The existing unsealed access road to the Maud Creek site into Katherine, heading south from the mine was used during previous mine operations to haul material from site. The access road is linked to the Stuart Highway, a major sealed arterial route, by Ross Road, a sealed public road which is regularly used by horticultural landholders in the area. The access road crossed Gold Creek just south of the existing mine pit and during the wet season, the road is often impassable and experiences significant damage.

To facilitate the transport of material from site, this existing access track would need to be upgraded and have a crossing installed over Gold Creek to enable transport operations to continue during the wet season both into and from site. This route has been approved by the Northern Territory Government as a Right of Way and General Service easement.

A second approved Right of Way and General Service easement runs north from the Mine to Gorge Road. This provides the site with a second access point to Katherine.

A possible new access route runs south west from the mine pit and joins a short unnamed gazetted road before exiting onto the Stuart Highway. The new route is approximately 4.5 km shorter than the existing road and remains exclusively on Lot 4192. Although this route may represent some savings in overall haulage costs, it is not an existing easement. Approvals will be required and significant heritage and environmental concerns would need to be addressed and therefore has been disregarded at this time as a viable alternative transportation route.

The proposed option is to upgrade the southern access track that connects to Ross Road.

Once on the Stuart Highway, the proposed haulage route heads North through the township of Katherine to the Ping Que Road turnoff. The proposed haulage route is presented in Figure 18-11. At the next level of design, a more detailed road and service provision can be made with further detailed information regarding the flood plans near the Gold Creek.

The budget cost to upgrade the current track with a provision for flood protection is AUD2.2M. An additional 20% contingency has been applied due to the level of energy undertaken. This cost is in included in the capital cost estimate in Section 21.

Figure 18-11: Haulage route from Maud Creek Mine to Union Reefs Processing Plant

18.7.1 Haulage Operations and Mine Traffic

It is anticipated that the mine haulage operations will utilise quad semi-trailers, which will involve travel from Maud Creek to Union Reefs (loaded) and back again (empty) on a daily basis; therefore, the mining operations will require approximately 26 road train movements per day (13 full and 13 empty). A summary of the haulage logistics is shown in Table 18-9.

A typical loaded quad semitrailer from Maud Creek mine site will have 22 axles, and a total/payload weight of about 160/ 110 tonnes respectively and complies with the Department of Transports maximum mass regulations. Confirmation of allowable number of trailers and load through town centres is required at the next phase of the pre-feasibility study. Any reduction will require a corresponding increase in truck movements each day.

Table 18-9: Haulage Logistics

Description	Units	Value
Annual Throughput	t/y	500,000
Number of shifts per day	No	1
Shift duration	h	12
Operating hours per day	h/d	10.8
Operating days per year	d/y	360
Operating hours per year	h/y	3,888

Description	Units	Value
Daily capacity required (average)	t/d	1,389
Truck payload (quad)	t/truck	110
Return trips per day	trips/d	13
Return trip distance	km/trip	288
Return trip duration	h/trip	4.73
Return trips per truck per day	trips/d/truck	2.28
Trucks required	**No**	**6**
Truck separation time	min	55

Based on 227.8 km @ 90km/hr, 60km @ 50km/hr, 1-hour loading/unloading per trip

Site supplies will be transported to the Maud Creek mine site from Darwin, Pine Creek area, Katherine or from South and Western Australia as required.

Staff commuting to the mine site will mainly be travelling from Katherine, via cars and minibuses, with commuting occurring across a 24-hour period throughout the life of the mine. Approximately 15 small vehicles including one or two minibuses are likely to be used daily to transport employees to site.

18.7.2 Impacts on Traffic

A summary of the existing traffic and the traffic volumes that are expected from the proposed mining activities including various types of vehicles on Stuart Highway are presented in Table 18-10.

The traffic volumes are based on the Department of Transports published traffic data at various locations along the proposed haul route. It can be seen from the above traffic volumes that there is a higher volume of traffic north of Katherine than south of it. This is due to the additional traffic between Katherine and Darwin and also the traffic travelling through Katherine using the Victoria Highway.

It is proposed that haulage from the site and traffic to the site will occur only during daylight hours. This would nominally be from 6:00AM to 6:00PM. This will occur throughout the year during both the wet and dry seasons. The proposed quad road train and light vehicle movements have been combined with the 2014 traffic numbers in Table 18-10 to assess the impact on traffic numbers.

Table 18-10: Stuart Highway Traffic Counts and Classification

Description	Units	Vehicle Class					Total
		Short Vehicle	Short Towing Vehicle	2, 3, 4 Axle Trucks	3-6 Axle Articulated Vehicle	B Double, Double & Triple Road Trains	
		1	2	3-5	5-9	10-12	
Stuart Highway - 30 km south of Katherine							
2014 Traffic count figures	%	89.8				10.2	100
	No.	853				97	950
Proposed new traffic	No.	40				26	66
New total	No.	893				123	1016
Percent increase	%	4.7				26.9	6.9
Stuart Highway - 20 km north of Katherine							
2014 Traffic count	%	92.9				7.1	100

Description	Units	Vehicle Class					Total
		Short Vehicle	Short Towing Vehicle	2, 3, 4 Axle Trucks	3-6 Axle Articulated Vehicle	B Double, Double & Triple Road Trains	
		1	2	3-5	5-9	10-12	
figures	No.	1383				106	1489
Proposed new traffic	No.	10				26	36
New total	No.	1393				132	1525
Percent increase	%	0.7				24.6	2.4
Stuart Highway - 2km North of Kakadu Hwy (near Unions Reefs)							
2014 Traffic count figures	%	90.1				9.9	100
	No.	1033				113	1146
Proposed new traffic	No.	10				26	36
New total	No.	1043				139	1182
Percent increase	%	1.0				22.9	3.1
Stuart Highway - 100m South of Victoria Hwy within Katherine urban zone							
2014 Traffic count	No.						9179
Proposed new traffic	No.	40				26	66
New total	No.						9245
Percent increase	%						0.7

2014 Traffic count figures taken from the Department of Transports “ *Annual Traffic Report 2014”*

Overall the increase in traffic on the Stuart Highway resulting from the mining operations at Maud Creek would be 2.4 percent north of Katherine, 6.9 percent south of Katherine and 0.7 percent within the Katherine Township. A graphical representation of the increased traffic numbers is provided in Figure 18-11.

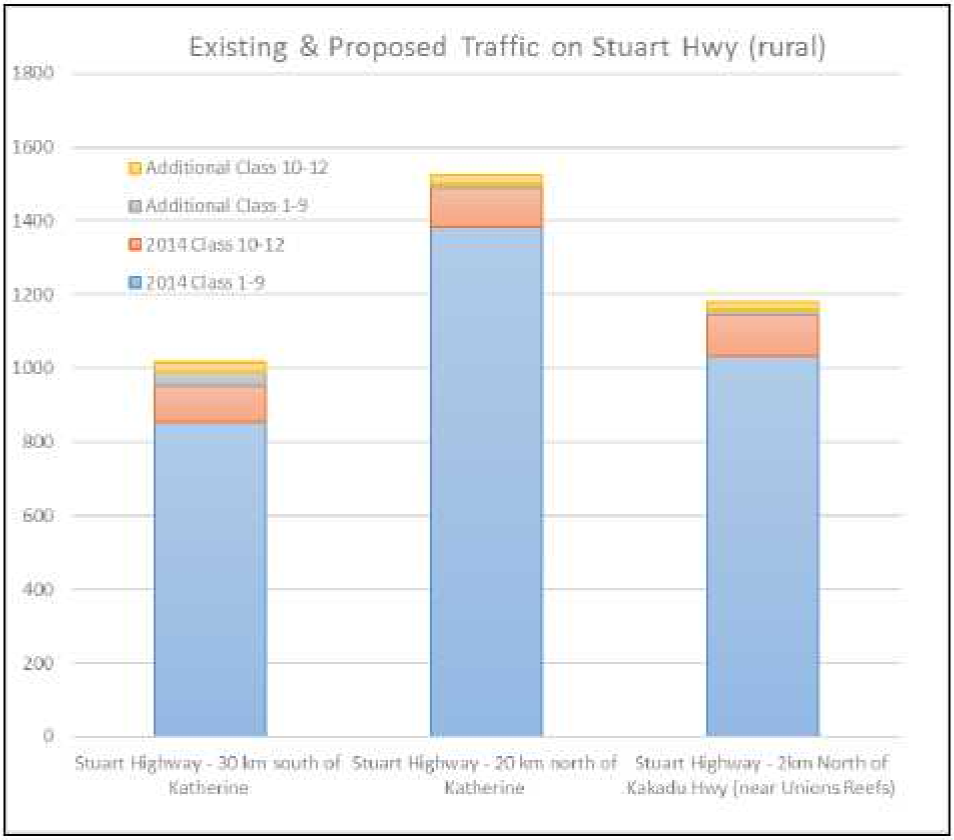


Figure 18-12: Impacts on traffic near Katherine and Union Reefs

This number of small vehicles represents less than a 5 percent increase in the existing traffic levels, which is considered minor and is not anticipated to pose a significant safety or traffic issue for other drivers in the area.

The 26 quad road train movements per day represents a 24.6 percent increase north of Katherine and a 26.9 percent south of Katherine. The major safety impacts of these road trains will be at the two intersections where the road trains turn onto or off the Stuart Highway and ensuring adequate overtaking opportunities for cars.

The Ross Road and Stuart Highway intersection currently has a left turning lane into Ross Road for unloaded road trains returning from the Union Reefs Processing Plant. The Ping Que Road and Stuart Highway intersection currently has an additional lane on the Stuart highway to allow traffic to move past a laden road train turning right into Ping Que Road. In future studies, a more detailed assessment of these intersections will be required to assess the adequacy and the level of safety the turning lanes provide.

Heavy vehicle route maps published by the Department of Transport show that the current Katherine road train route follows the Stuart Highway through the centre of township. As shown in Table 18-10, the percentage increase in traffic through the township is less than 1 percent. Speed limits through the town are set at 50 km/h to reduce safety risks to pedestrians and small vehicles.

In July 2015, the Northern Territory Government started the planning on a long-term heavy vehicle alternate route that bypasses the central business area of the township. The alternate heavy vehicle route study was proposed following a number of road safety issues in the main street, as it currently accommodates a mix of pedestrians, local and tourist traffic, and heavy vehicles. The alternate heavy vehicle route would dramatically reduce the impact of the Maud Creek haulage operations. Government and public consultation and approval will be a key element to the viability of processing of Maud Creek mineralization at the Union Reefs processing Plant.

18.8 Concentrate Transport

The flotation gold concentrate is loaded into 1.5 t capacity bulk bags, which are then loaded into shipping containers ready for transportation. The containers are loaded onto a road train that transports the concentrate by road from the Union Reefs Processing Plant to the Darwin Port. Railway transport has not been considered due to the need for a siding, capital cost, the relatively small volume of concentrate, short LoM and proximity and connectivity to Darwin via a major highway.

Costs were requested from Northline for the overland transport of the containers and have been included in the operating cost.

18.9 Site Facilities

Diesel Storage

Self-bunded diesel storage tank facilities of 80,000 - 100,000 litres will be located at the mine. This diesel storage caters for all underground and surface diesel needs as well as emergency black start fuel for back diesel generators for critical processing equipment.

Maintenance Facilities

A surface maintenance workshop facility will be located in the vicinity of the mine to service and maintain the underground fleet and associated mining equipment. All servicing and maintenance activities are undertaken on surface (i.e. none underground).

A maintenance/ boilermaker workshop will be located at the Processing Plant to assist with undertaking the required maintenance activities.

18.10 Housing and Land

The closest centre of population to the Maud Creek Gold Project is Katherine, which is a regional centre that enjoys excellent infrastructure, services and communications. The Airport at Katherine is serviced by Airnorth and has regular flights to and from Darwin.

Based on preliminary discussions with Newmarket Gold, it has been assumed that the workforce will consist of local/relocated residents, drive in/drive out from Darwin and other local towns such as Pine Creek and Adelaide River, and drive-in drive-out personnel to supplement the overall workforce in equal proportions.

Rented housing and hotel accommodation is available for mining personnel in Katherine. The Ibis Styles Hotel is located to the east of Katherine on the Stuart Highway and has approximately 100 rooms.

The 112 person Pine Creek camp is located approximately 120 km from the Maud Creek site and is currently in care and maintenance. The commute time to this camp makes it unsuitable for housing workers from the Maud Creek Mine.

The Cosmo accommodation village is used to house the Cosmo Underground Mine employees and contractors as well as the Union Reefs Processing Plant which is approximately 53 km south. This is also available for the very small increase in numbers required for processing and haulage contractors.

19 Market Studies and Contracts

19.1 Marketing

The third party toll treatment options assessment was based on indicative terms provided to Simulus Engineers for the sale of a gold concentrate to Shangdong Zhong Guo (China Gold Shandong) for processing at their Yantai Gold Smelter. Similar terms were provided by Baxville (Beijing) Minerals Trading Ltd.

MRI Trading AG has also been approached to provide indicative terms for the Maud Creek concentrate. They are facilitating the sale of KBL' s Mineral Hill gold concentrate. Dialogue continues but they have not yet provided final confirmation of pricing. Verbal confirmation of terms has been provided by KBL and has been used to benchmark the terms selected as the base case assumption.

Other inquiries have been issued to potential customers and traders. Indicative terms for the sale to another direct exporter of concentrate (i.e. to consolidate with their concentrate) who sell to Guoda Gold Co Ltd, or sale to an Australasian POX facility with additional capacity have been referenced from other similar gold concentrates projects and operations in Australia. Informal discussions have also been held with the owners of a number of Australian third party refractory gold operations for sale of a generic gold concentrate and to assess available capacity at those facilities but discussions have not been progressed to any great extent. It has been used mainly in the process of elimination of the bulk of these options. Further investigations into the remaining options can be undertaken at the next stage of study if deemed appropriate.

The sale of the gold concentrate to China remains the current base case. No formal concentrate discussions or sales contracts have been entered into at this stage of study.

Gravity gold will be recovered into gold doré on site and sold to established Australian refiners.

19.2 Contracts

No formal concentrate sales contracts are required at this stage of study, nor have any been arranged.

20 Environmental Studies, Permitting, and Social or Community Impact

20.1 Environment and Social Aspects and Impacts

20.1.1 Social and Economic Context

The Maud Creek Project lies within the Town of Katherine local government area (LGA 72200), which occupies an area of some 7417 km^2 (Figure 20-1) and within the broader Katherine region (336,674 km^2, Figure 20-2). The traditional owners of the land, the Jawoyn people, have occupied the Katherine region for thousands of years.

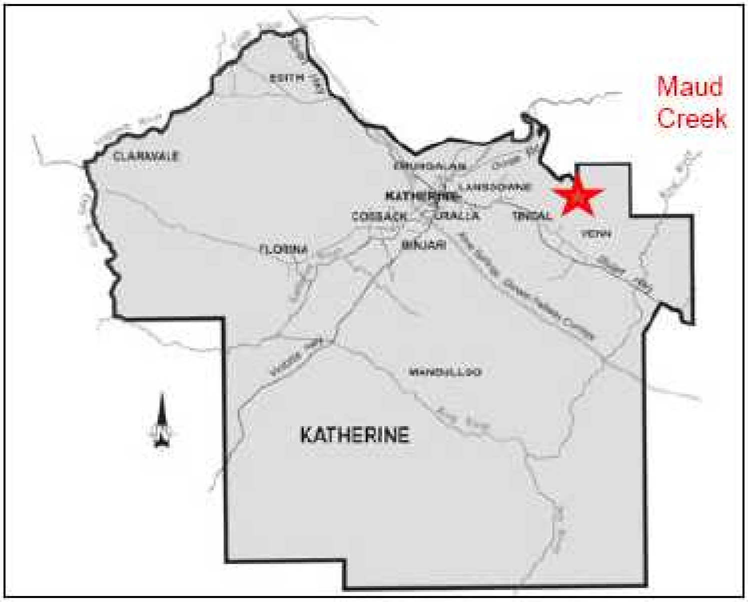


Figure 20-1: Katherine municipality local government area (ABS, 2011 census)

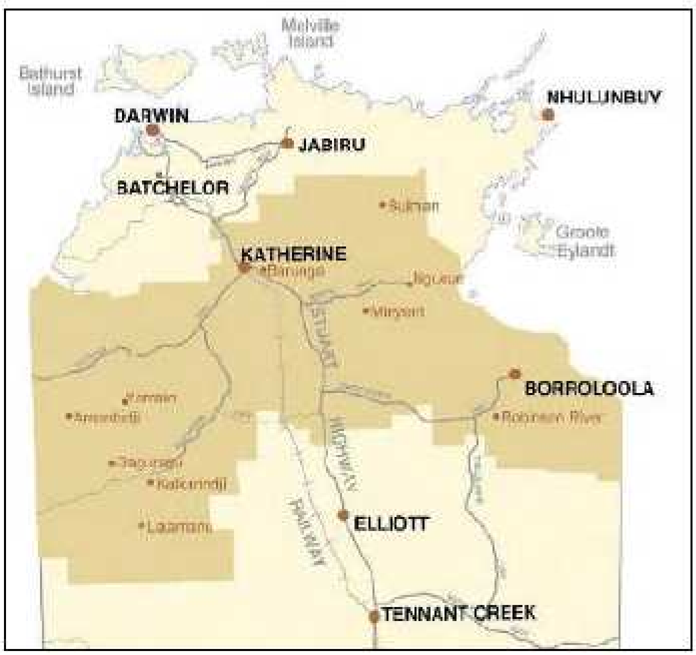


Figure 20-2: Katherine region (shown in darker tan)

Source: Katherine Land Use Plan, 2014

The town of Katherine is the fourth largest population centre in the Northern Territory, after Darwin, Palmerston and Alice Springs. The town was formally gazetted in 1926. At the 2011 census, the population of the Katherine municipality was estimated at slightly under 11,000 people, of whom about 24% identified as indigenous (Aboriginal or Torres Strait Islander). In younger age groups, the percentage of Aboriginal people is higher (Figure 20-3). Compared to the Australian population as a whole, the Katherine community is relatively young, with a median age of 31. The Katherine population is characterised by a high degree of mobility, with a relatively large proportion of the population changing place of residence (either within the region or interstate) between consecutive censuses.

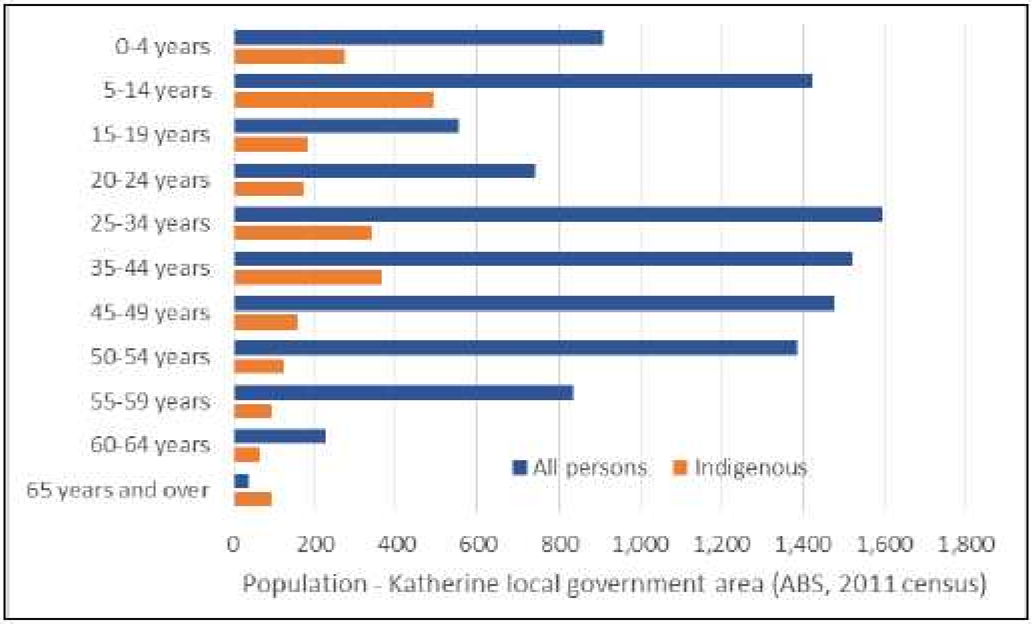


Figure 20-3: Katherine LGA population, by age group (2011 census)

Katherine is an important regional centre, providing government services (health, education, transport, communications and business development functions) to the wider region. A range of basic utilities, services and community infrastructure is available, including:

A 60- bed hospital; various community and private medical clinics;

A range of public and private education and training providers from pre- primary to university level; an airport (shared with the RAAF Base Tindal);

Police, fire and emergency services;

Water supply, sewerage and waste disposal facilities, and

Cultural, sport and outdoor leisure facilities, such as the Nitmiluk National Park.

The modern economy of the Katherine region has traditionally been dominated by the agricultural and pastoral sectors, but in the past few decades mining, public administration and safety (including defence), tourism and construction (largely related to major resource and infrastructure projects) have also been important contributors to the regional economy. Although mining has been by far the greatest contributor to gross regional product in the past few years (Figure 20-4), it is not a major employer in the Katherine municipality (Figure 20-5).

Unemployment and labour force participations rates in Katherine are, on average, similar to those in Australia as a whole, although unemployment among young people (those aged 24 years and below) is conspicuously higher than average unemployment rates in the general Australian community (Figure 20-6 and Figure 20-7). In 2011 about 41% of the Katherine population aged 15 and over had some type of post-secondary school qualification, mostly at Certificate or Diploma levels.

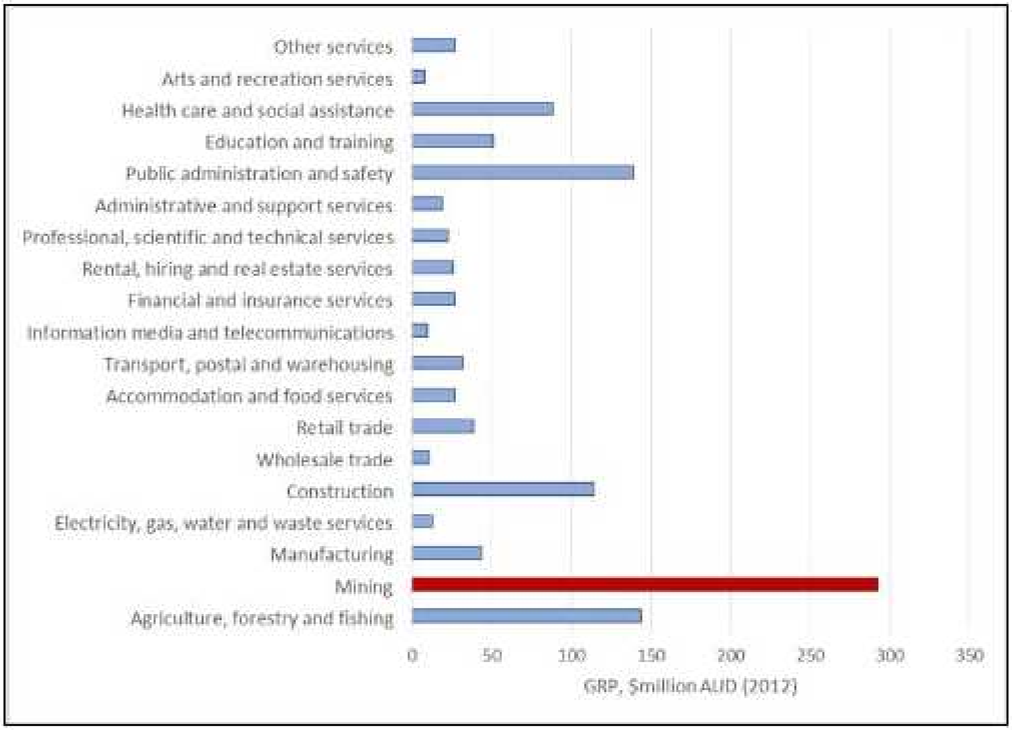


Figure 20-4: Gross regional product, by industry – Katherine NT (2012)

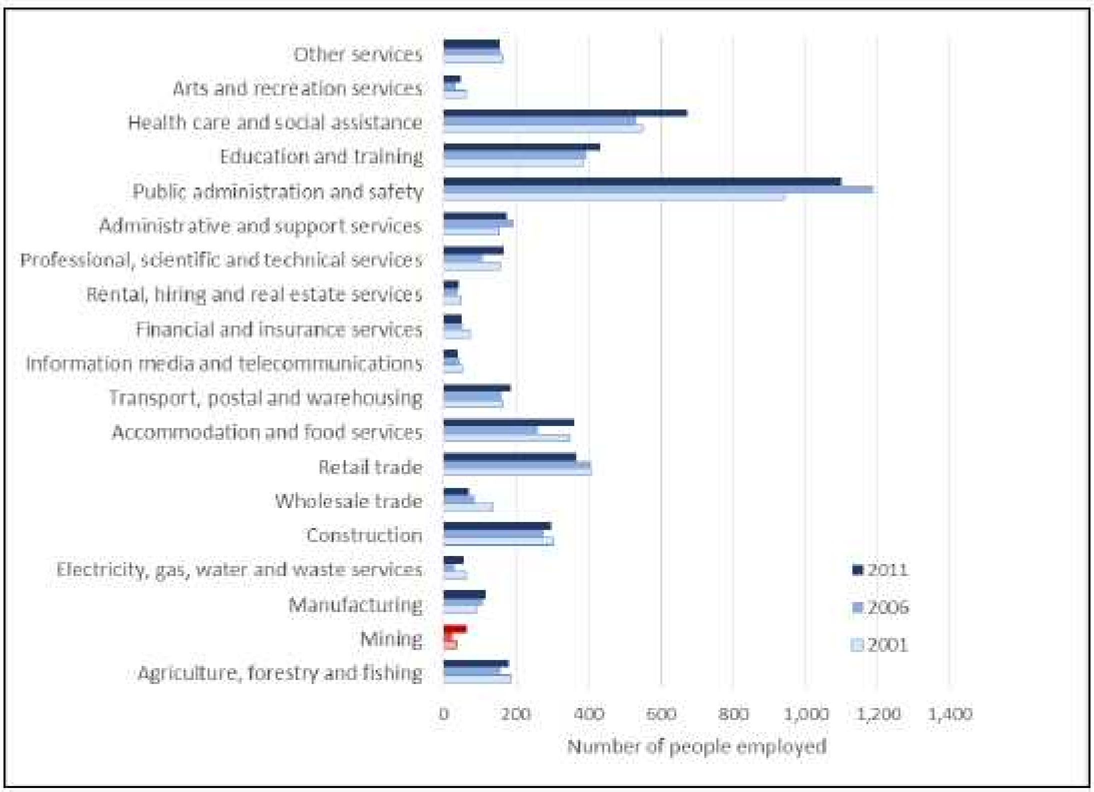


Figure 20-5: Employment by industry sector – Katherine LGA

Source: ABS data

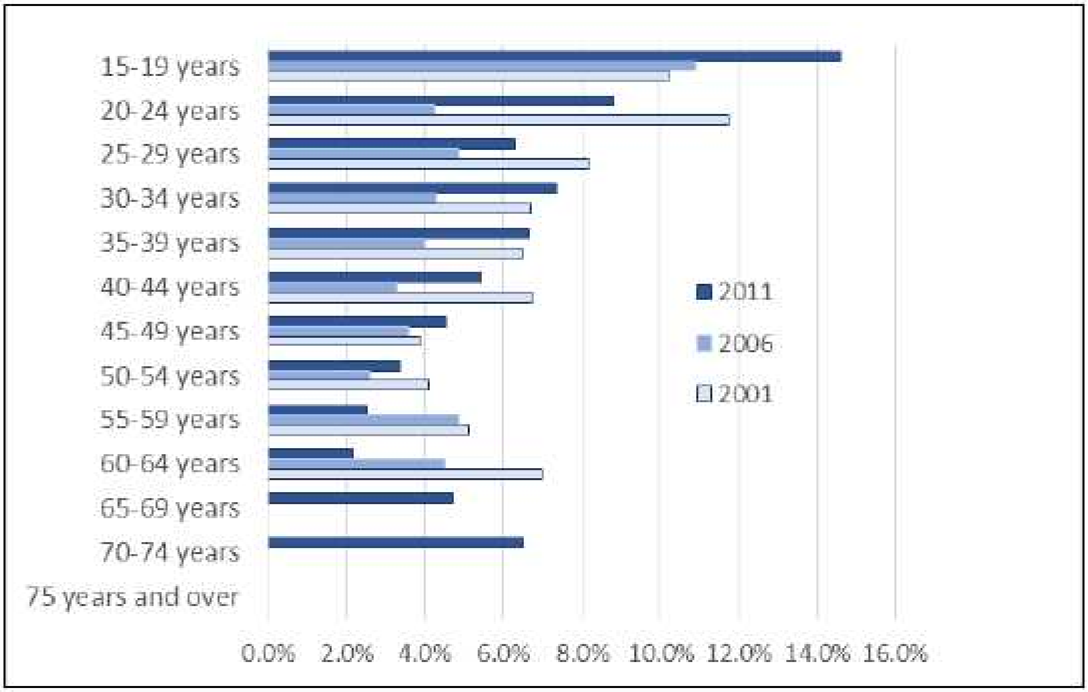


Figure 20-6: Unemployment rates – Katherine LGA
Source: ABS data

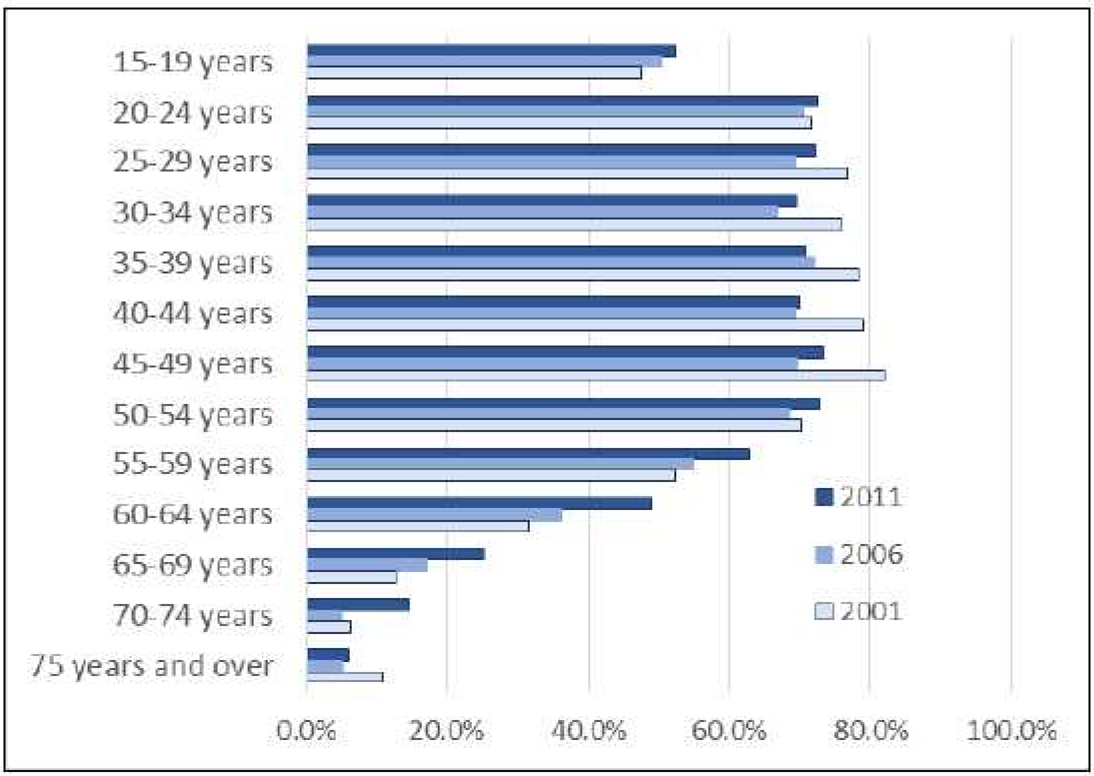


Figure 20-7: Labour force participation rate, Katherine LGA (ABS data)

Median weekly household income reported at the last census in the Katherine LGA was approximately AUD1,424. Median weekly rental and mortgage repayments at the 2011 census were AUD200 and AUD433, respectively. Current median rentals range from around AUD320/ week for a unit to AUD450/ week for a house (https://www.realestate.com.au/neighbourhoods/katherine-0850-nt, accessed 14 Jan 2016). Slightly over 50% of the Katherine population lives in rented accommodation.

Approximately 3,860 private dwellings exist in the Katherine municipality, of which about 8.5% (329 dwellings) were unoccupied at the time of the 2011 census. Housing types include single dwellings (~58%); semi-detached structures, townhouses, units and apartments (~17%); and a range of other accommodation, such as caravans, tents and improvised accommodation. Northern Territory government budget forecasts predict continuing strong demand for housing in Katherine, with relatively low vacancy rates (NT Government budget, 2013-2014). The Town of Katherine Land Use Plan (2014), anticipates the need for an additional 81ha of urban residential land and 180 ha of rural lifestyle lots (rural, rural living and rural residential) to accommodate residential development to beyond 2026. In the main, the areas being considered for additional residential development lie to the south and northwest of the Katherine urban centre and are unlikely to encroach on the Maud Creek project area, however evolving land use patterns may need to be taken into account as part of transport planning for the project.

The Katherine community comprises a number of different groups, including non-resident tourists and seasonal workers, part-time residents (FIFO workers), employees of the RAAF Base Tindal and their families, and full time residents. Stakeholder engagement for the Maud Creek project will need to take account of these diverse stakeholders, as well as the needs and expectations of Aboriginal traditional owners and their representative bodies, along with the requirements of government stakeholders at local, regional and territory level.

20.1.2 Surface Water

The Maud Creek Project area is traversed by Gold Creek, an ephemeral tributary of Maud Creek, which flows into the Katherine River upstream of the water supply extraction point for the Katherine Township. The distance between the proposed mine operations area and the confluence with the Katherine River is approximately 10.5 km. The Katherine municipality sources its water from a combination of groundwater bore and the Katherine River. The town water supply use of water from the Katherine River is sometimes constrained by the high turbidity in surface watercourses during the wet season. Neither Maud Creek nor Gold Creek contribute significant (or possibly, any) flow to the Katherine River during the dry season, although remnant pools persist along the watercourses throughout the dry season. Hydrological modelling conducted in connection with Terra Gold' s Maud Creek proposal (discussed in Section 4.5) concluded that the channels of both Gold Creek and Maud Creek would be likely to experience over bank flows on several occasions in every wet season. Modelling conducted by SRK is consistent with the earlier assessment (Figure 20-8).

Surface water quality in the project area has been characterised to a limited extent. Dry season sampling conducted by previous tenement holders between 1968 and 1998 recorded generally good quality water, of near-neutral pH, relatively low salinity and low concentrations of most trace metals (URS, 2008). The EIS prepared in connection with the Terra Gold Maud Creek proposal suggested that surface waters in the project area may occasionally show elevated concentrations of copper or lead, presumably as a result of mineralization in the catchment, however additional monitoring would be required to demonstrate this convincingly.

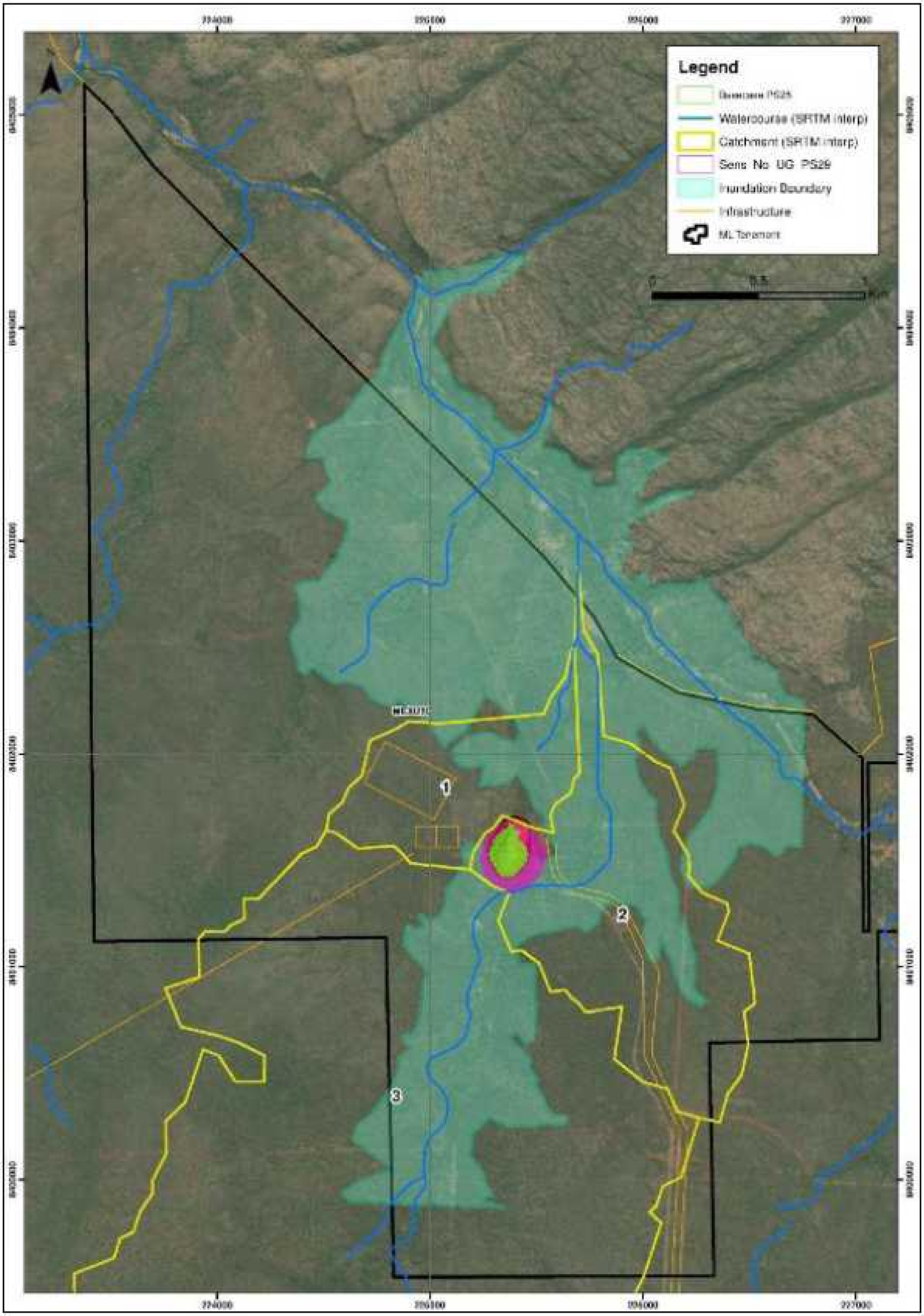


Figure 20-8: Estimated flood extents in absence of engineering controls

20.1.3 Groundwater

Groundwater beneath the Maud Creek site occurs primarily within fractured rock (Tollis formation) aquifers and unconsolidated alluvial sediments. Groundwater recharge is expected to occur from infiltration of rainfall, especially in areas of outcropping quartz / quartz breccia. Some recharge would also occur seasonally through the bases of watercourses. The proposed mine operations area lies to the north of the Tindall limestone formation, upon which the Town of Katherine, and a range of local enterprises, rely for water supply (Figure 20-9).

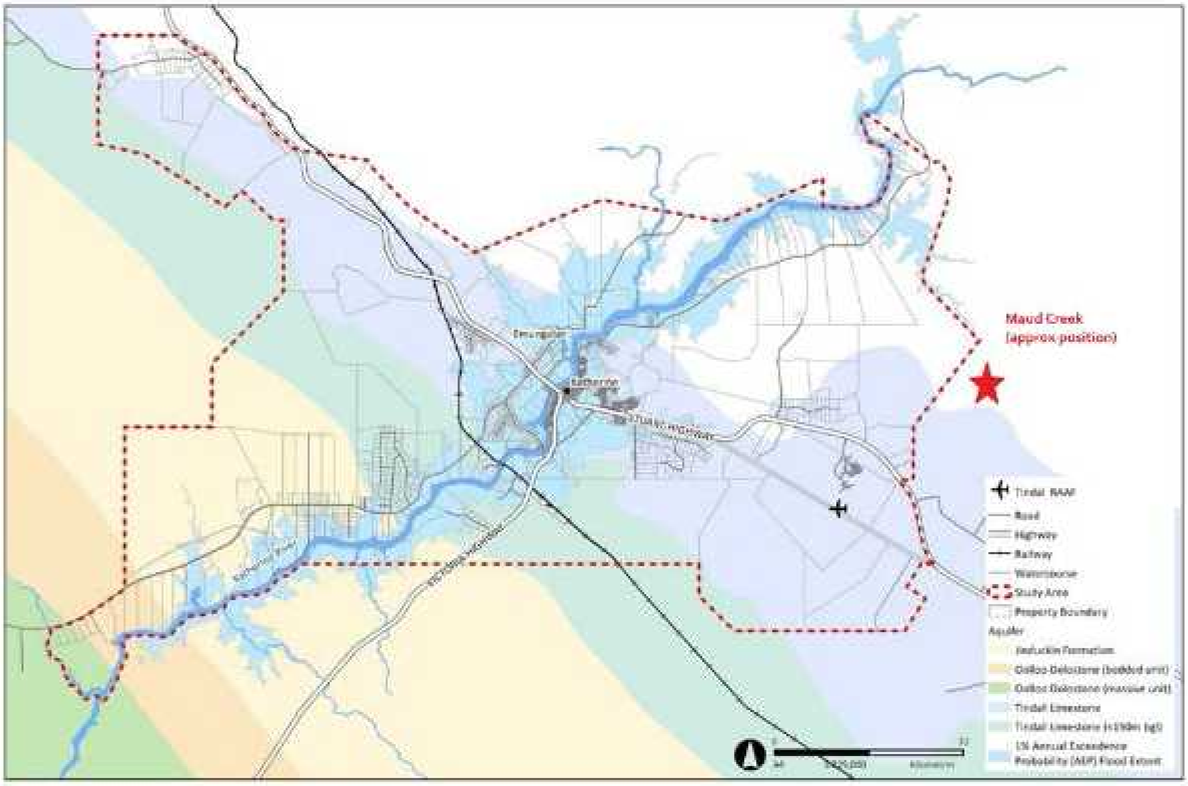


Figure 20-9: Groundwater aquifers near Katherine

Source: Katherine land use plan, 2014

The groundwater table in the Maud Creek area generally lies at shallow depth (between 1 and 6m below ground surface, corresponding to an average elevation of approximately 141m AHD). The seasonal variation in the depth to water is expect to be in the range of 2 to 4m. The general direction of groundwater flow is to the northeast, away from the Tindal limestone aquifer.

Baseline monitoring conducted in connection with Terra Gold' s EIS (URS, 2008) reported the dry season groundwater level at the confluence of Gold and Maud Creeks to be approximately 6.9 m below ground level (approximately 4 m below the creek bed). Limited if any contribution of groundwater to local stream flow is likely during the dry season, as the groundwater levels will generally lie below the beds of the watercourses.

Groundwater quality is typically fresh, with total dissolved solids concentrations below 600 mg/L. Groundwater chemistry is dominated by calcium and magnesium bicarbonates and the groundwater pH is slightly alkaline, with an average pH around pH8. Information in Terra Gold' s Maud Creek EIS suggested that the local groundwater may have naturally elevated concentrations of arsenic and/or selenium, however this conclusion appears to be based on limited water quality testing and would need to be confirmed through further groundwater monitoring.

20.1.4 Native Flora and Vegetation

As part of the EIS, Terra Gold commissioned field surveys of flora and vegetation for its Maud Creek proposal. The botanical investigations, which were carried out in March and April 2007, identified 153 plant species of which 19 were introduced weeds. None of the 153 plant species recorded during the 2007 were listed under the Commonwealth Environment Protection and Biodiversity Conservation Act 1999 (EPBC Act) as vulnerable, threatened, rare or endangered. One species, Tephrosia humifusa, recorded at 5 sites during the field surveys, is listed as Near Threatened (NT) under the Territory Parks and Wildlife Conservation Act 2006. The 2007 survey reported noted that Tephrosia humifusa appeared to be common locally, occurring at over one quarter of the floristic study sites and in a range of different habitat types. At the time of the Terra Gold EIS, Tephrosia humifusa was known to occur at 5 different locations in the Northern Territory. The Maud Creek survey records represented a range extension of the species. Current government records shown 19 occurrences of Tephrosia humifusa in the Northern Territory (not including at Maud Creek, refer Figure 20-10). The plant remains on the Northern Territory endangered species list (2012).

Figure 20-10: Distribution of *Tephrosia humifusa*

Source: Atlas of Living Australia, accessed 15/01/2016

Northern Territory government databases show a number of protected flora species in area to the north of the Maud Creek project area, but none within the mine tenement ((data from http://nrmaps.nt.gov.au/) (Figure 20-11).

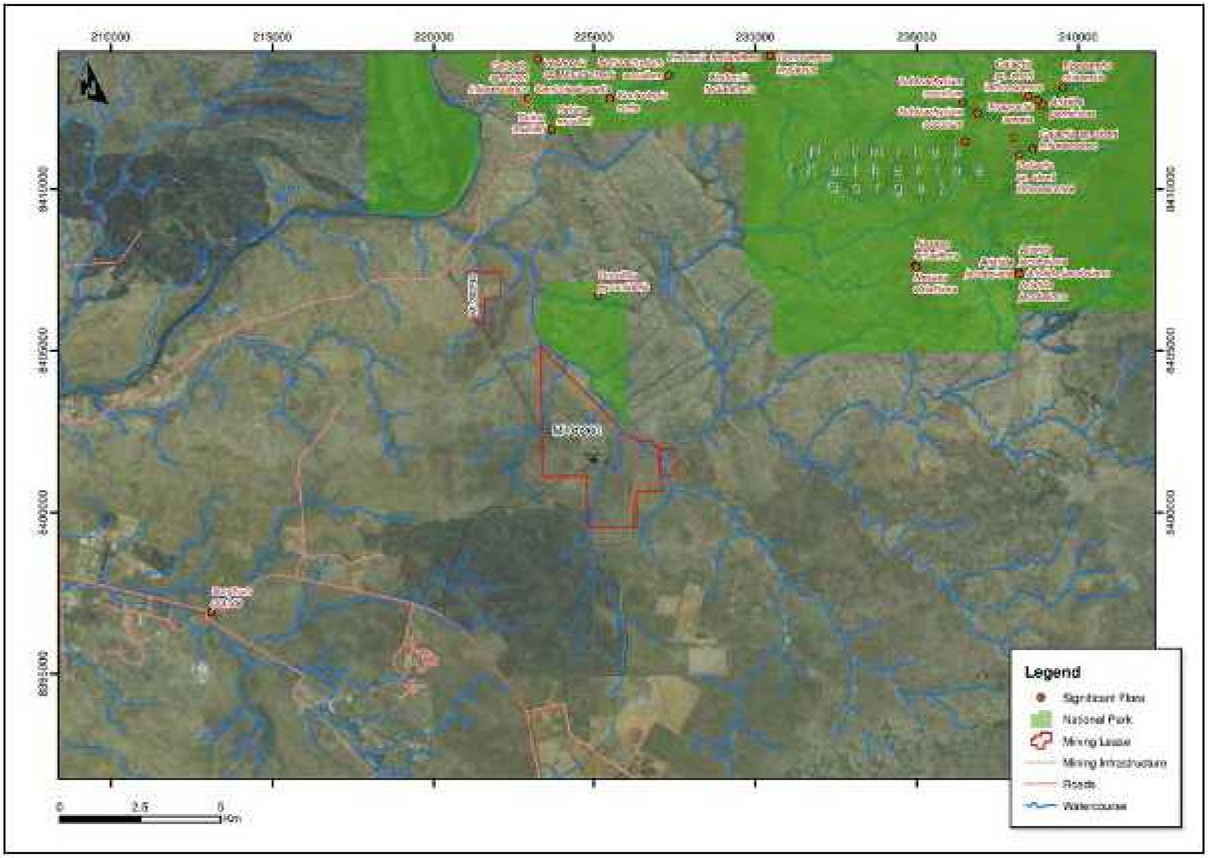


Figure 20-11: Conservation significant flora
Source: http://nrmaps.nt.gov.au/)

Ten distinct vegetation communities were mapped in the Maud Creek Project area during the 2007 surveys (Figure 20-12). The vegetation communities included open forest, woodland and open woodland assemblages. None of the vegetation types was considered of special conservation significance: all occur widely outside the project area and none is thought to support specially protected flora or fauna species. No vegetation communities protected under the EPBC Act were recorded during the 2007 surveys. There is currently no mechanism for listing Threatened Ecological Communities under NT legislation. However, one threatened ecological community that occurs in the general project locality is listed as Endangered under the Commonwealth EPBC Act. This is the Arnhem Plateau Sandstone Scrubland Complex, which has had protected status since 2011. The Arnhem Plateau Sandstone Scrubland Complex is known to occur in parts of the Nitmiluk (Katherine Gorge) National Park. It not likely to occur within the Maud Creek mine tenement. This would need to be confirmed as part of future project assessment and permitting.

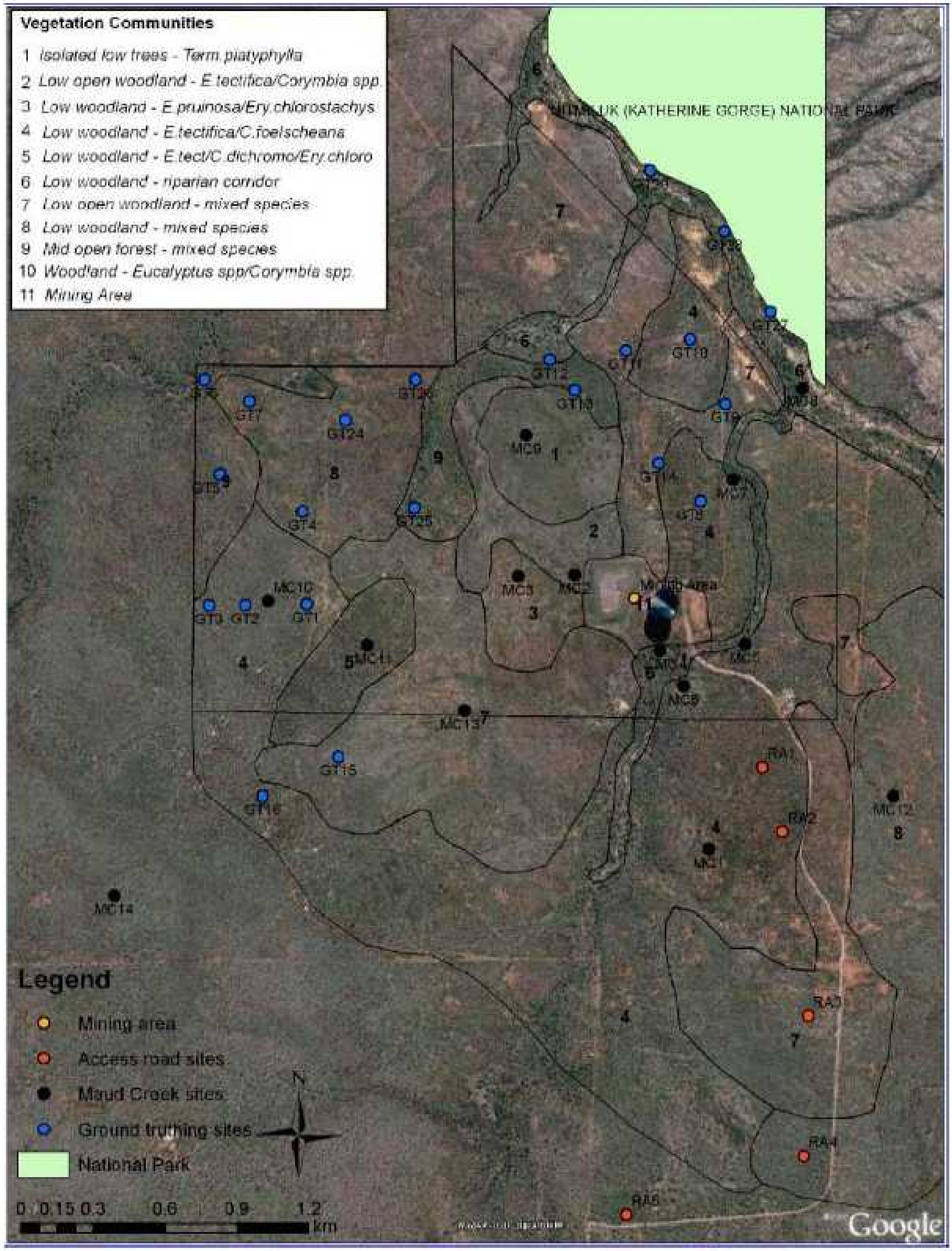


Figure 20-12: Vegetation communities
Source: Crawford & Metcalf, 2007

20.1.5 Native Fauna and Habitats

Terrestrial fauna habitats

The most recent field fauna surveys of the Maud Creek project area were conducted in April and May 2007. Previous surveys had been carried out in 1994, 1996 and 1997 (EMS, 2007). The 2007 surveys recorded a range of habitat types, including open woodland, grassland and riparian habitats. Distinctive and geographically limited habitat types underlain by limestone karst or limestone outcrops were noted to the west of the mine tenement, in an area that had been proposed for an access road. No limestone outcrop was recorded within the mine tenement itself. Overall, the fauna habitats within the proposed mine operations areas were not characterised by significant biodiversity. Many areas had been disturbed by past mining activities, grazing pressure, weed incursion and disturbance from swamp buffalo, cattle, pigs, feral donkey and fires. Riparian habitats, which are not locally extensive, have been degraded by large numbers of swamp buffalo. Habitats within the mine tenement are unlikely to support significant biodiversity. None of the habitats present in the project are likely to provide significant habitat for water birds or shore birds. The limestone outcrop habitats to the west and south of the mine operations areas are less heavily impacted and may act as locally significant refuge and areas that support conservation significant wildlife, including bats and a variety of invertebrate species such as land snails.

Terrestrial vertebrate fauna

A total of 144 native and six introduced terrestrial vertebrate species were recorded during the 2007 surveys. These included 30 mammals, 91 birds, 18 reptiles, and 11 amphibian species. A number of varanid (monitor lizard) species observed during previous surveys of the area were absent during the 2007 surveys. This may be the result of predation and displacement by cane toads, which were introduced to the area in about 2001. Cane toads were the most common amphibian observed during the surveys.

The majority of fauna recorded or expected to occur in the Maud Creek Project area are widespread in northern Australia. Two bird species of conservation significance – the red goshawk (*Erythrotriorchis radiatus*) and the Australian bustard (*Ardeotis australis*) have been observed within the mine tenement. Both of these are relatively widespread within the NT. Six species that are considered Near Threatened under the according to the Territory Parks and Wildlife Conservation Act 2006 were also recorded in or very near to the proposed mine operations area ((data **from http://nrmaps.nt.gov.au/)**

Figure 20-13). They included: the bush stone-curlew (*Burhinus grallarius*), hooded parrot (*Psephotus dissimilis*), grey falcon (*Falco hypolelucos*), northern nailtail wallaby (*Onychogalea unguifera*), Arnhem sheathtail bat (*Taphozous kapalgensis*), orange leafnosed bat (*Taphozous georgianus*) and western chestnut mouse (*Pseudomys nanus*). No significant migratory bird species were recorded on or near the mining tenement. (The EPBC-listed rainbow bee-eater (*Merops ornatus*) was recorded, but this species is very common and is unlikely to be significantly impacted.)

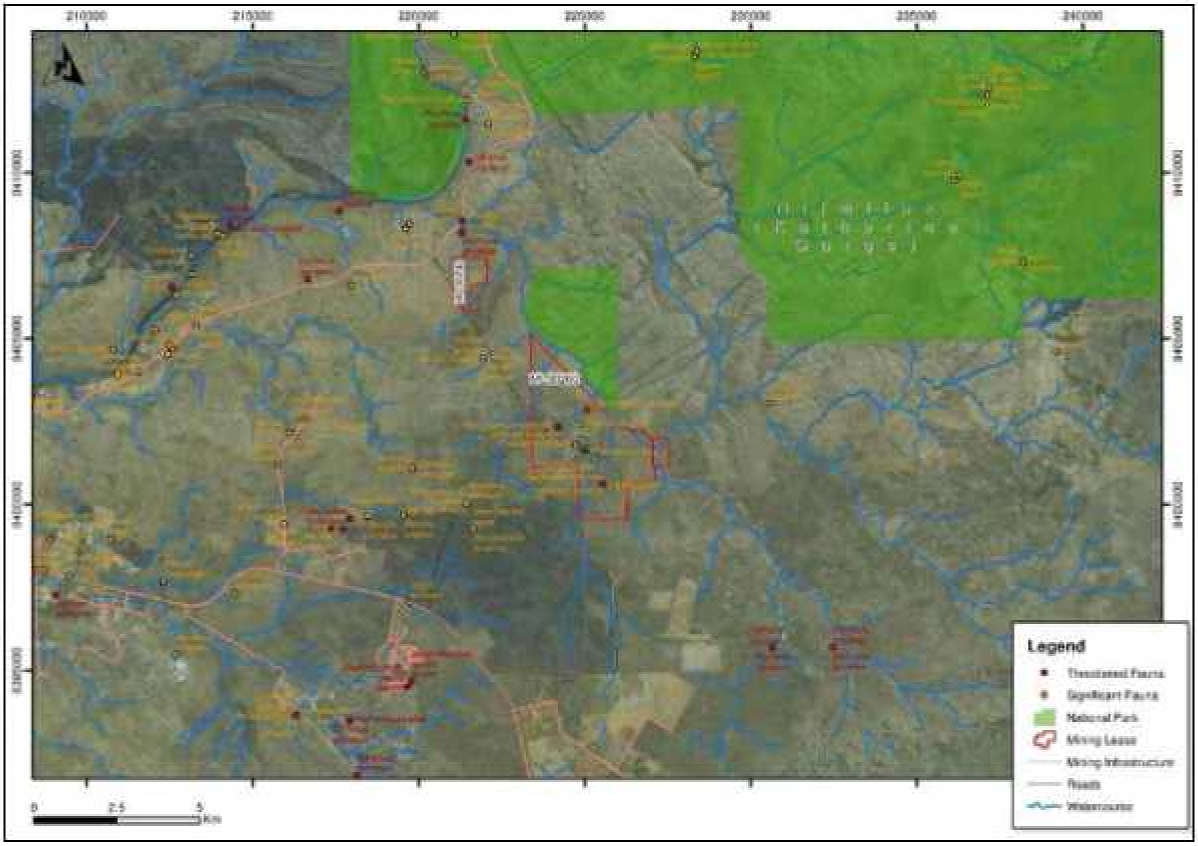


Figure 20-13: Conservation significant fauna

Source: http://nrmaps.nt.gov.au/)

Terrestrial invertebrate fauna

Land snails were collected and identified at all the limestone outcrop areas surveyed to the west of the mine tenement (along the access road alignment previously proposed by Terra Gold). Some of the snails observed in the limestone outcrop habitats (*Xanthomelon* sp) were also collected from open forest habitats in the project area. This species is thought to be relatively widespread, but occurs at low densities within its range (EMS, 2007). The most common snail species collected was *Torresitrachia weaberana,* a moderately commonly species known to occur in areas between Katherine and Kununurra. One of the snail species collected was an undescribed genus and undescribed species that is known to be limited in range to limestone karst in the Tindal – Cutta Cutta region. This species was be considered to be of regional conservation significance. Recent technical studies on snails in the Katherine region have proposed that endemic land snails can be used as bio-indicators for the health of some ecological assemblages and may themselves be under threat of extinction (Braby *et al*, 2011, Willan *et al*, 2009).

Aquatic fauna and habitats

Fifteen species of freshwater fishes were recorded in Gold Creek and Maud Creek during the 2007 field surveys. The western rainbowfish (*Melanotaenia australis*) was the most abundant species in samples and was the only species present at all sites. Other common species included the banded grunter (*Amniataba percoides*), spangled grunter (*Leiopotherapon unicolor*) and northern trout gudgeon (*Mogurnda mogurnda*) (URS, 2008).

No listed threatened aquatic species were recorded in recent or previous field surveys in the project area. Although the EPBC Act lists the freshwater sawfish (*Pristis microdon*) as a vulnerable species that could potentially occur in the vicinity of the project area, no suitable habitat for this species occurs in proximity to the proposed mine operations it. Neither is the species known to occur in the Katherine River.

None of the freshwater fishes recorded during past surveys of the Maud Creek project area are listed as threatened in the NT or under Commonwealth legislation. The species occurring Maud Creek are all common and widespread forms that are well adapted to the variable instream conditions that characterise the system Aquatic habitat quality along Gold Creek was described as ‘significantly impaired’ by trampling and other disturbance by swamp buffalo, donkeys, feral pigs and cattle. Neither Maud Creek nor Gold Creek are considered likely to support high levels of biodiversity in the project area, although the riparian habitats may afford a level of shelter and temporary feeding and growing habitat. The watercourse may provide a migration path for aquatic species during the wet season (URS, 2008).

20.1.6 Air Quality, Noise & Vibration

The proposed mine operations area is located at least 5 km from the nearest ‘sensitive receptor’ (residential premises, for example). Providing conventional environmental controls on dust and noise emissions are implemented during construction and operations, it is unlikely that significant impacts on environmental amenity will arise.

There is some risk that offsite traffic associated with the project will give rise to public concern. Arrangements for controlling impacts of product haulage, workforce traffic and vehicle movements for transport of fuel and reagents will need to be described as part of the project’ s environmental impact assessment.

20.1.7 Conservation Areas

The closest conservation area to ML30260 is the Nitmiluk (Katherine Gorge) National Park, owned by the Jawoyn Aboriginal Land Trust and jointly managed with the Parks and Wildlife Commission (Figure 20-14). The park is notable for its scenery, cultural values and ecological integrity. The proposed mining operations are unlikely to give rise to direct impacts on Nitmiluk National Park, although there is some potential for indirect impacts on the park, particularly if the Maud Creek Project includes the upgrade and use of the general service easement running north from the mine to Gorge Road. Increased use of the northern access route has the potential to increase the risk of spreading weeds and/or pathogens and may exacerbate the adverse environmental impacts caused by animal pests. Improved road access could also increase the risk of bushfires, by making the area generally more available to the public.

It is unlikely that the mining operations would intrude on parts of the park that are routinely accessed by visitors, although some consideration of light spill and noise (blasting, offsite traffic) may be required as part of a future environmental impact assessment.

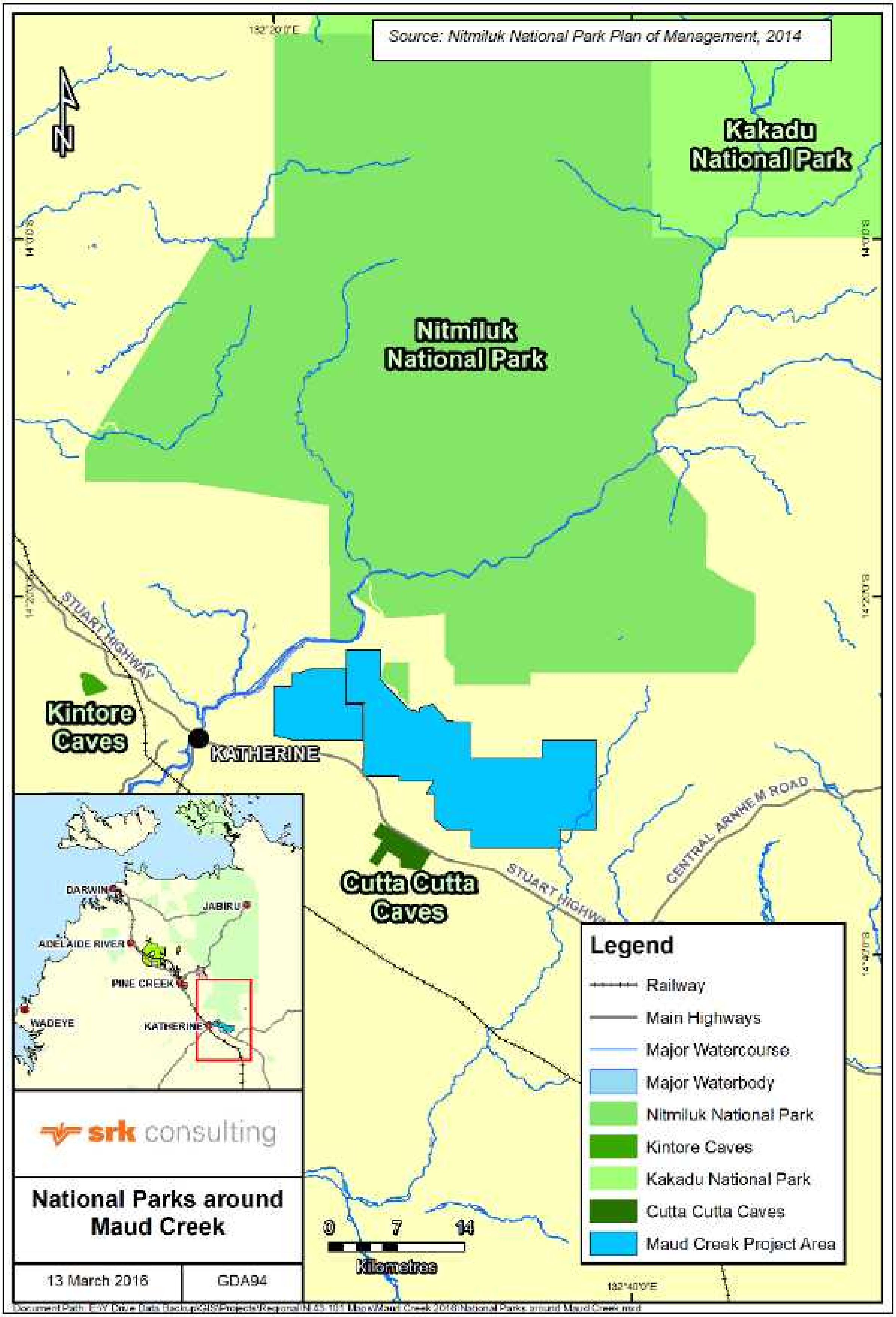

Source: Nitmiluk National Park Plan of Management, 2014
Kakadu National Park
Nitmiluk National Park
Kintore Caves
KATHERINE
STUART HIGHWAY
CENTRAL ARNHEM ROAD
Cutta Cutta Caves
DARWIN
JABIRU
ADELAIDE RIVER
PINE CREEK
WADEYE
KATHERINE
Legend
Railway
Main Highways
Major Watercourse
Major Waterbody
Nitmiluk National Park
Kintore Caves
Kakadu National Park
Cutta Cutta Caves
Maud Creek Project Area
srk consulting
National Parks around Maud Creek
13 March 2016
GDA94
0 7 14
Kilometres
132°40'0"E

Figure 20-14: Locations of parks and reserves near Maud Creek

20.1.8 Heritage Values

Heritage surveys relating to both Aboriginal and non-Aboriginal values of the Maud Creek mine site were conducted as part of baseline studies for the Maud Creek EIS prepared by Terra Gold (URS, 2008, refer Figure 20-15).

The studies identified sixteen Aboriginal archaeological sites, including 14 stone artefact scatters and three stone quarries (one the quarries was associated with an artefact scatter). Each site was rated as having high, moderate or low significance, based on rarity, intactness and age of the site. In all, over 240 stone artefacts were identified. Sixteen of the artefact scatters were considered to have high significance, three were classified as having moderate significance and five were described as having moderate to low significance. Two of the three stone quarries were described as highly significant.

A number of additional sites were identified on the Maud Creek tenements subsequent to the completion of Terra Gold' s EIS. The current Newmarket Gold Cultural Heritage Environmental Management Plan (2015) identifies 18 Aboriginal sacred sites. Disturbance of any Aboriginal sacred site would require formal authorisation (refer Section20.2 ..8).

Two non-Aboriginal heritage sites were identified during baseline surveys for Terra Gold' s Maud Creek project. One site comprised a series of historic alluvial gold mine diggings at the eastern side of the project area. The site was described as having significant heritage value. The second non-Aboriginal heritage site was a feature from an historic settlement to the northeast of the project area. The site consists of a raised stone hearth and a foundation made from cement, gravel and earth and edged with cobbles of rock. A variety of other artefacts were identified (shards of stoneware and earthenware, preserved meat tins, tobacco tins and matchboxes). These were generally rated as having a moderate to low significance.

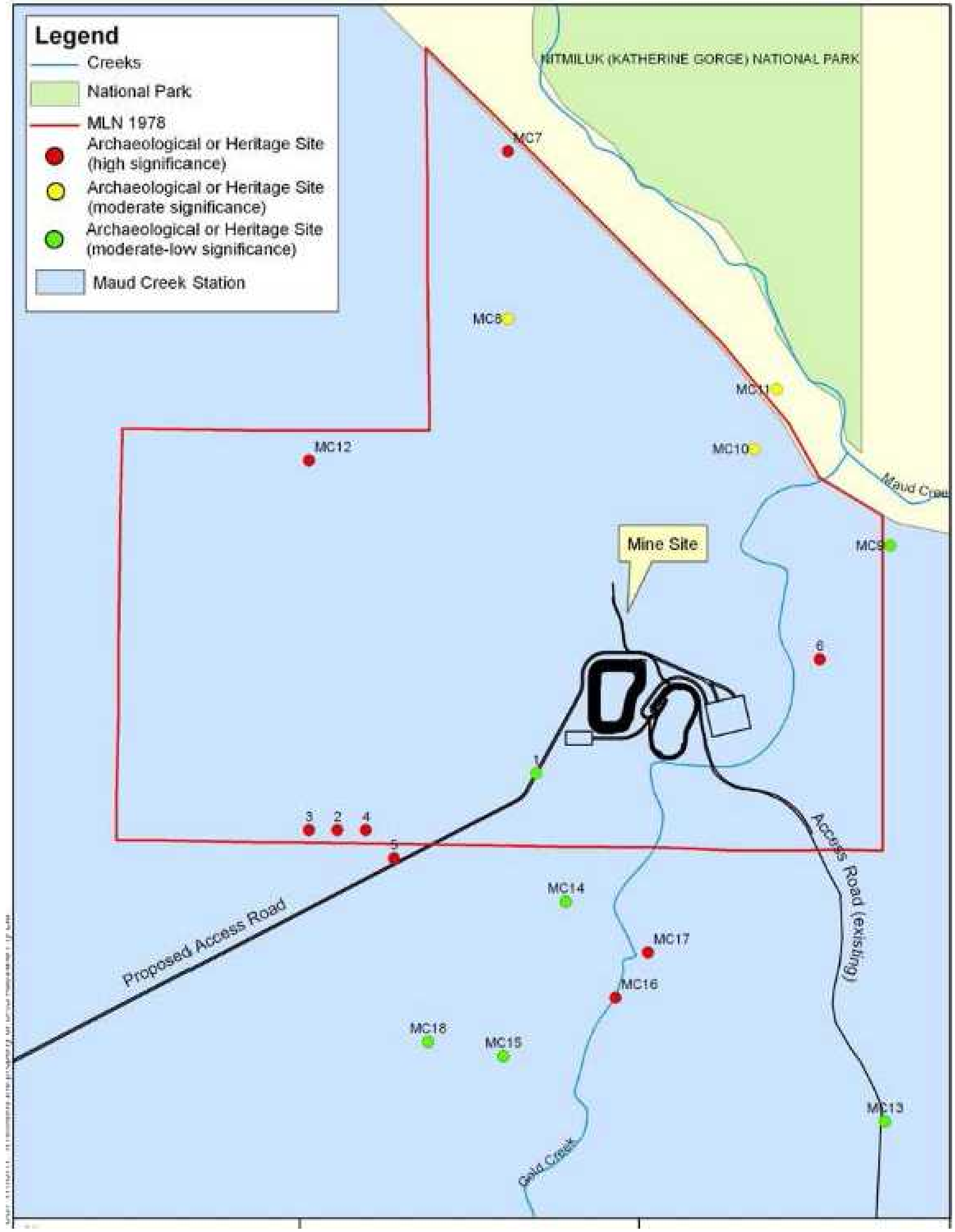


Figure 20-15: Aboriginal and non-Aboriginal heritage sites

Source: Figure 11.1 from URS, 2008

20.1.9 Mine Closure and Revegetation

The EIS prepared for Terra Gold' s Maud Creek Project included a preliminary discussion of mine rehabilitation objectives and outcomes. The approach proposed by Terra Gold was to rehabilitate disturbed land to allow future use primarily for pastoral purposes, with some areas (such as waste rock dumps) potentially allocated for conservation purposes. It was suggested that water in pit voids might be used for watering livestock. The closure strategy also proposed to use pit voids for disposal of reactive materials remaining at the ROM. No closure provision was made for rehabilitation of tailings or waste rock storage facilities, as the project did not include these elements.

A significant review of mine rehabilitation and closure will be required as part of project assessment and permitting, both to reflect changing community expectations and to align with evolving government regulations and policy. Long-term management of mine wastes and of water in pit voids is likely to be a central issue for mine closure design. Newmarket Gold should expect an increased focus on verifiable rehabilitation performance targets.

20.1.10Potential Impacts

Although a range of environmental impacts is possible as a result of implementing the Maud Creek Project, the aspects most likely to attract the attention of stakeholders are:

Protection of surface water and groundwater quality, especially from contaminants contained in tailings and other mineral wastes;

Avoidance of culturally significant areas;

Potential for impacts on public safety and/or amenity from mine traffic in or near the town of Katherine

Control of indirect impacts (weeds, fire, feral animals) on the environmental values of Nitmiluk National Park

Newmarket Gold' s ability to deliver acceptable mine rehabilitation outcomes

A summary of project aspects and impacts is presented in Table 20-1.

A more detailed analysis and formal risk assessment of these aspects and impacts will be required as part of project assessment and permitting.

Table 20-1: Preliminary aspects and impacts analysis

	Social / economic	Public health & safety	Aboriginal heritage	Surface water	Groundwater	Soil quality / land capability	Flora / vegetation	Faun
Land access & clearing for mine development			**X**	**X**			**X**	
Land access & clearing for access road(s)			**X**	**X**			**X**	
Mine dewatering	**X**			**X**	**X**			
Mine operations: haulage &								

traffic within mine site								
Mine operations: offsite haulage & traffic	**X**		**X**					
Blasting, excavation, heavy equipment operation								
Handling and stockpiling of waste rock				**X**	**X**	**X**		
Ore processing				**X**	**X**			
Tailings storage	**X**			**X**	**X**			

Establishment and operation of mine accommodation	**X**							
Waste generation (non- process wastes)				**X**	**X**			

20.2 Regulatory Approvals

20.2.1 Mineral Titles Act

With the exception of certain prescribed substances (such as uranium), all minerals located in the Northern Territory (except for certain prescribed substances including uranium) are the property of the Northern Territory Crown. The Mineral Titles Act took effect in November 2011, replacing the Mining Act. The Mineral Titles Act provides a framework for granting and transferring mineral titles to authorise exploration for and extraction of minerals. It also establishes a basis for authorising ancillary activities related to mining.

The Mineral Titles Act recognises seven categories of mineral title, of which the most relevant to the Maud Creek Project is a mineral lease (ML). Newmarket Gold was granted a mineral lease (ML30260) over the project area on 14 April 2014. The lease over the 106 ha tenement is valid until 13 April 2024 and can be extended, subject to certain conditions (including expenditure requirements and compliance with tenement conditions).

Grant of a mineral lease gives the tenement holder exclusive rights for the development of mineral projects on land within the title boundaries. Subject to the tenement holder obtaining an authorisation under the Mining Management Act, the grant of a mineral ease confers rights to conduct mining and processing of minerals, as well as a range of mining related activities, including:

- exploration for minerals;
- treatment of tailings
- storage of waste

Additionally, the Mineral Titles Act provides the holders of mineral titles strong rights of access and important entitlements related to the taking and use of water. For example, a mineral title holder may apply for an access authority to enter land which is outside the title area for the purpose of constructing infrastructure or for the taking of water.

The Mineral Titles Act also provides a basis for assigning liability for mine rehabilitation. An entity taking ownership of a tenement on which disturbance exists (as is the case at Maud Creek) assumes the risks and responsibility for that tenement, irrespective of whether it causes any additional mining disturbance to the land.

20.2.2 Mining Management Act

Before any mining activity can occur on a granted mining title the intending operator of the mining activity must apply for and be granted an authorisation under the Mining Management Act. The Mining Management Act (MMA) effectively constrains rights conferred by the Minerals Title Act (MTA) by prohibiting mining activities until the intending operator (which may or may not be the tenement holder) has:

- been granted an authorisation (supported by an approved mining management plan (MMP)), and
- lodged a security to cover the full costs of rectifying any environmental harm arising from mining activities and for final rehabilitation of the affected area (including rehabilitation of any legacy disturbance).

Authorisations granted under the Mining Management Act typically cover all mining (and related) activities required for project implementation. In some circumstances, separate authorisations may be granted for activities (operation of an explosives storage facility, accommodation village, power station) which the mine operator does not itself wish to operate and manage.

Mining management plans (MMP) are binding legal instruments through which pollution and environmental harm are controlled on mine tenements. Operators of mining projects are exempt from some licensing requirements that would normally apply to comparable activities on non-mining land. The MMP instead serves regulate resource usage (taking of water) and other environmental aspects of mining (and related activities) within the bounds of the granted tenement. It is an offence to contravene the environmental obligations arising from an approved MMP, if the activities result in environmental nuisance, harm or pollution. More generally, the Act prohibits the release of wastes or contaminants unless the release is done in accordance with an approved MMP.

20.2.3 Waste Management and Pollution Control Act 2009

The Waste Management and Pollution Control Act is administered by the NT EPA. The Act regulates the collection, transport, storage, treatment and disposal of listed wastes. Wastes – including both process and non-process wastes – arising from mining activities are generally exempt from licensing requirements under this Act and are instead regulated through the approved mining management plan.

20.2.4 Water Act 1992

Mining operations in the Northern Territory are not required to seek licences for abstraction and use of water under the Northern Territory Water Act, providing their use, storage and management of water is done in accordance with an approved mining management plan. Mine tenement holders are entitled to construct bores, and to take or divert water (not including water in dam or wells belonging to others) and to use the water for mining and mineral processing.

Pollution provisions of the Water Act generally do not apply to mining operations, except in the event that contamination or pollution moves beyond the tenement boundary (either through an intentional or accidental discharge). Mine operators are exempt from discharge licensing requirements that would normally apply under the Water Act, to the extent that:

- the contaminant or waste results from carrying out of a mining activity;
- the waste or contaminant discharge is confined within the land on which the mining activity is carried out; and
- the discharge is done in accordance with an approved mining management plan.

In circumstances where a discharge of a waste or contaminant (for example, water from mine dewatering) is likely to move beyond the boundaries of the mine tenement, a discharge licence would be required. As an example, Newmarket Gold holds a waste discharge licence (WDL 138-02) to authorise release of wastewater from its Union Reefs site to Wellington Creek and the McKinlay River. Newmarket Gold has implemented comprehensive environmental management systems for several of its NT operations: similar arrangements would be appropriate at Maud Creek.

A waste discharge licence may be required for activities that could result in seepage that could cause pollution of an aquifer. The implications of this for storage of tailings and waste rock and for the development of permanent pit voids would need to be discussed with NT regulators.

20.2.5 Aboriginal Land Rights (Northern Territory) Act 1976 (Cwlth)

Contemporary use of the term ‘traditional owner’ largely derives from the Aboriginal Land Rights (Northern Territory) Act 1976 (ALRA). The ALRA established ways for Aboriginal people to claim land in the territory on the basis that they were the traditional Aboriginal owners of the land. Further, the ALRA gave traditional Aboriginal land owners the right to control access to their land, including the right to withhold consent for exploration activities on their land. This Act has limited, if any, relevance to the Maud Creek Project, as mining tenure has already been granted and the project lies on freehold land that is not subject to the control of any Aboriginal Land Trust.

20.2.6 Native Title Act 1993 (Cwlth)

The Native Title Act 1993 establishes a legal framework for the recognition and protection of native title. It also establishes an administrative system for:

Determining claims to native title;

Validating past acts which might otherwise be invalidated because of the existence of native title; and

Developing processes and standards for future dealings which could affect native title.

The Maud Creek Project is located on freehold land, and therefore is not exposed to native title claims. There are no registered or determined native title claims over the project area.

20.2.7 Heritage Act 2011

The *Heritage Act 2011* declares all Aboriginal archaeological places and objects heritage places. Other historically or culturally significant places (for example, buildings and heritage objects) are also protected under the *Heritage Act* if they are listed on the NT Heritage Register. No non-Aboriginal heritage sites formally registered on the Australian Heritage database are known to occur in the project area. No formal authorisation would be required in relation to disturbance of the non-Aboriginal heritage sites and Terra Gold made no commitments to the protection or conservation of these sites.

The Act makes provision for approval of works to salvage Aboriginal archaeological sites, subject to the consent of the appropriate Traditional Owner or Site Custodian. It is normally a requirement that appropriate studies be conducted to assess the character, extent and value of a site before salvage. A person proposing to disturb a heritage site would be expected to commit to a program to prevent or minimise damage and to make provision for suitable curation of any artefacts. For Aboriginal sites, suitable curation typically involves return of the object(s) to the Traditional Owners or protection if artefacts are outside of the operational footprint.

20.2.8 Northern Territory Aboriginal Sacred Sites Act 1989

The *Aboriginal Sacred Sites Act* provides the legal basis for protection of all Aboriginal sacred sites, irrespective of whether the site is formally registered. Carrying out work – or even accessing – a sacred site requires a consent in the form of a certificate issued by the administering agency, the Aboriginal Areas Protection Authority (AAPA).

A certificate (reference number D89/199:90/804 (Doc: 68552) C2009/266 (supersedes C2007/072)) was issued by the AAPA to Crocodile Gold on 8 October 2009. The certificate authorises access to / disturbance of nominated Aboriginal sites within the Maud Creek mine site (ML30260 – formerly MLN1978), as well as some other sites on other Newmarket Gold tenements named on the certificate. The consent is subject to a range of implementation conditions, including a condition excluding works or access to designated sites. No works of any kind are permitted at sites SS5369-69 (restricted work area 1), 5369-32 (restricted work area 2) or 5369-27. Two of the exclusion areas (restricted work areas 1 and 3) lie ML30260, refer Figure 20-16.

A further implementation condition (Condition 3) says, *this certificate shall lapse and be null and void, if the works in question or the proposed use is not commenced within 24 months of this certificate*. Newmarket Gold has advised that some of the works approved under the AAPA certificate (specifically, exploration drilling) was carried out in 2011, prior to the expiry of the AAPA consent. Accordingly, the consent is still valid.

With regards to the NT Aboriginal Sacred Sites Act, Newmarket Gold have the requisite permit (AAPA Authority Certificate C2009/266 superseding C2007/072), provided the project footprint remains within the Subject land as defined in the Authority Certificate, and avoids the two exclusion areas, then no further permitting is required.

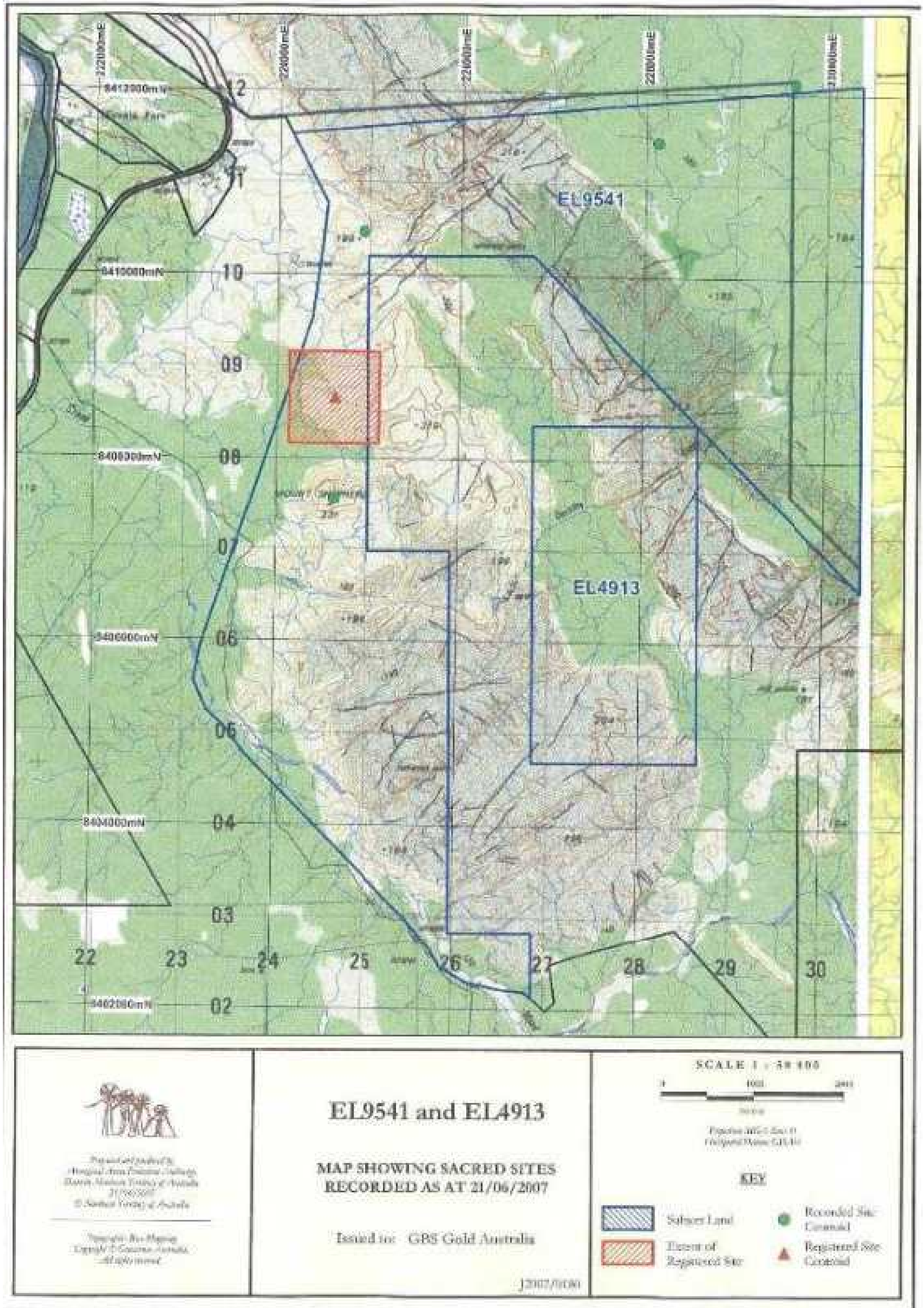


Figure 20-16: Restricted work areas

Source: Excerpt from AAPA certificate, June 2007

20.3 Waste Rock and Mineralization Geochemistry

20.3.1 Available Data

Drillhole Database

The current drillhole assay database (*Database_08_05_2015*) contains assay data for Au, As, Cu, Pb, Zn, Cr, and Ni. A smaller extended assay dataset was obtained from drilling in 2011, which includes Ca, Mg, Ag, Bi, Fe, Hg, K, Mn, Mo, Na, S, Sb, Sn, Ti and Zr.

Total sulphur can sometimes be used as an indicator of acid generating potential - the assumption being that all sulphur is present in the form of potentially reactive, acid generating sulphides.

The sulphur assay sampling density was low, with 2,140 samples collected from 21 drillholes (Figure 20-17) relative to the 2,129 drillholes included in the database.

The sulphur assay data was separated into waste-grade and ore-grade assuming a gold cut-off value of 0.1 mg/kg. Sulphur content in the waste rock (n=1,831) assays ranged from <0.05 -8.55%S (mean 0.16%S), while in the ore-grade assays sulphur ranged between <0.05 -10.7%S (mean 0.92%S) . In the waste rock lithologies the highest sulphur contents were present within the dolerite (8.55%S), sediments (SLST, 3.75%S), Cambrian Basalt (3.55%S), tuff (2.75%S) and veins (2%S).

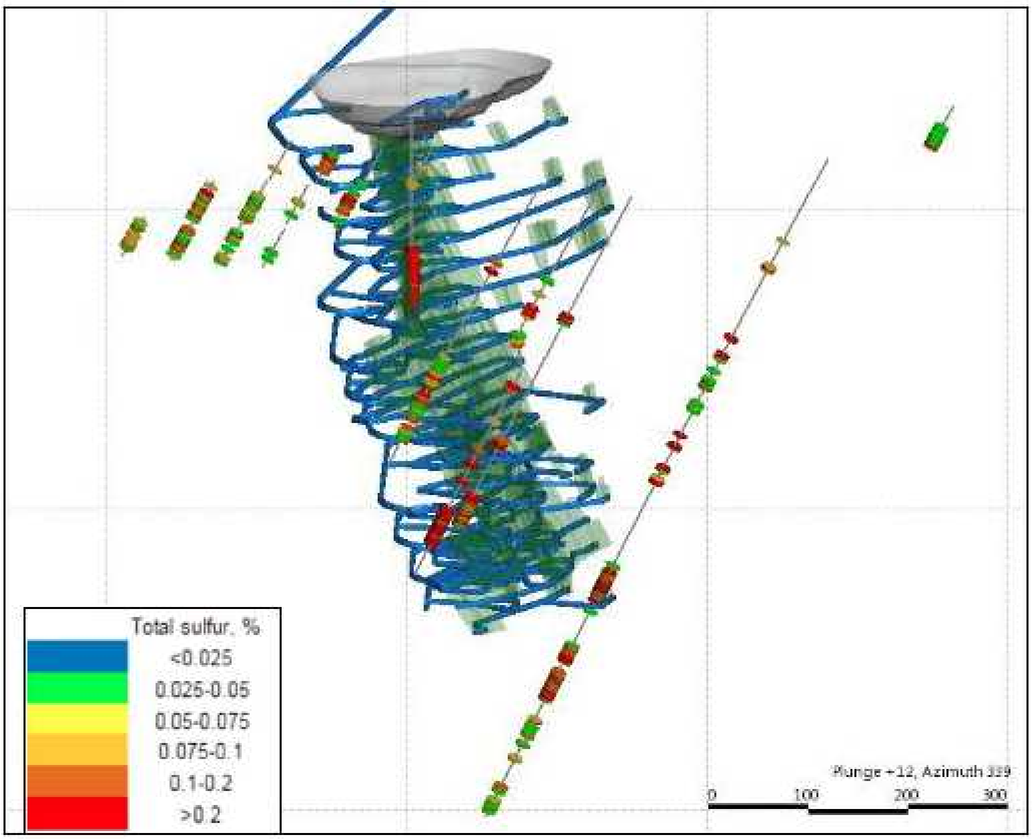


Figure 20-17: Drillhole database S assay data

Notes: The figure also shows the proposed open-cut pit (grey), underground workings (blue) and modelled deposit (green)

Detailed Geochemical Characterisation

Previous geochemical characterisation of waste rock (33 samples), mineralization (11 samples) from the weathered (or oxide) zone and the fresh (unweathered) zone, and soil (2 samples) from the Maud Creek deposit was undertaken in 1996 (Graeme Campbell & Associates, 1997). The assessment was prepared in support of an Environmental Impact Statement prepared by Kalmet Resources NL (Kalmet), to identify materials that may generate Acid Mine Drainage (AMD) and/ or impact water quality and revegetation.

The sampled materials included 33 waste rock samples, 11 mineralized samples and 2 soil samples. The waste rock and mineralization samples were collected from the weathered and fresh rock zones – a breakdown of the lithologies sampled is given in Table 20-2.

Table 20-2: Sampled materials (GCA, 1997)

Waste Rock/ Ore	Weathering Zone	Kalmet Lithology	SRK Lithology	No. of Samples
Waste Rock	Weathered Zone	Hanging wall Mafic Tuff	Tuff	10
		Footwall Sediments Undifferentiated	Sediments (SDST)	1
		Footwall Sandstone	Sediments (SDST)	3
	Fresh Zone	Hanging wall Mafic Tuff	Tuff	11
		Footwall Sediments Undifferentiated	Sediments (SDST)	4
		Footwall Sandstone	Sediments (SDST)	4
Mineralization	Weathered Zone	Stockwork Quartz - Tuff Hosted	Vein	1
		Stockwork Quartz - Sediment Hosted	Sediment (MSED)	1
		Massive Quartz - Tuff Hosted	Vein	1
	Fresh Zone	Stockwork Quartz - Graphitic Sediment Hosted	Vein (2), MSED (1)	3
		Stockwork Quartz - Tuff Hosted	Tuff	1
		Massive Quartz - Graphitic Sediment Hosted	Vein (1), Tuff (1)	2
		Massive Quartz - Tuff Hosted	Tuff (1), Vein (1)	2

The assessment included the following test work on all 44 samples:

pH and EC (paste/slurries; 1:2 solid : water)

Total S

Acid Neutralisation Capacity (ANC)[2]

Net Acid Producing Potential (NAPP)

The following additional testwork was carried out smaller sub-sets of samples:

Sulphate-sulphur (SO_4-S) and sulphide-sulphur (14 samples)

Net Acid Generation (NAG) (15 samples)

Multi-element analyses, (12 samples) (digest not specified) including the following elements: Ag, Al, As, B, Ba, Bi, C, Ca, Cd, Co, Cr, Cu, F, Fe, Hg, K, Mg, Mn, Mo, Na, Ni, P, Pb, Sb, Se, Sn, Sr, Th, Tl, U, V, Zn)

Saturation-Extract (SE) tests (7 waste rock samples) to assess the soluble salts content (Na, K, Mg, Ca, Cl and SO_4) and the solubility of arsenic and antimony.

20.3.2 Sample Representivity

The drill-hole assay samples with sulphur data are generally located in rock volumes that lie outside of proposed open pit and underground workings (Figure 20-17).

With respect to samples submitted for detailed characterisation, six of the uppermost samples occur within depths ranging from 2-3 m below ground level (mbgl) to 174-175 mbgl; and include 7 samples from the weathered (or oxide) zone and 19 from the fresh zone[3]. No samples have been collected below 175 mbgl.

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[2] Modified ANC test (unspecified number of tests on waste rock) using dilute sulphuric acid to measure readily available ANC (ANCC).

In conclusion, current datasets do not adequately represent the volumes to be mined as part of the currently proposed open pit and underground workings.

20.3.3 Geochemical Characteristics

Results generated from the detailed characterisation are summarised in Table 20-3. Note that, as discussed in the previous section, the samples studied do not give good representation of material that could be mined as part of currently proposed open pit and underground workings. However, the data obtained have value in that they can be used to infer some characteristics of materials located within the topmost 175m.

Most of the waste rock samples contained an excess of ANC and would be classed as NAF (Figure 20-17). Highest ANC was offered by the Hanging wall Mafic Tuff samples; however, note that only a portion (10-20%) of ANC was considered readily available for reaction. The balance of acid potential and neutralising capacity is closest in the case of the Footwall Sediments and Footwall Sandstone samples, including three samples that were classed as having uncertain acid generating potential and two samples that classed as PAF.

Multi-element analyses results were used to calculate Global Abundance Indices (GAI) to identify elements that are enriched relative to average crustal abundance concentrations. In both waste rock and mineralization, elements identified as being enriched included arsenic, antimony, chromium, nickel, tin, boron and silver.

Saturation-Extract (SE) tests (9 samples waste rock; 2 samples ore) gave alkaline extraction solutions (pH 8.6 -9.2) . Of the enriched elements listed above, only arsenic and antimony were included in the leach analysis suite – giving dissolved concentrations of arsenic up to 0.56 mg/L (one of the mineralization samples) and antimony up to 0.046 mg/L (one of the waste rock samples). The results suggest that both arsenic and antimony could be leachable from material to be mined at Maud Creek. [It is noted that the existing pit lake water was found to contain elevated arsenic concentrations (185-201 µg/L; relative to the ANZECC (2000) freshwater trigger value of 13 µg/L).]

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[3] More recent geological interpretation of the weathering zone extents would place 15 of the waste rock samples within the oxide (weathered) zone, 1 within the transitional zone and 17 in the fresh zone.

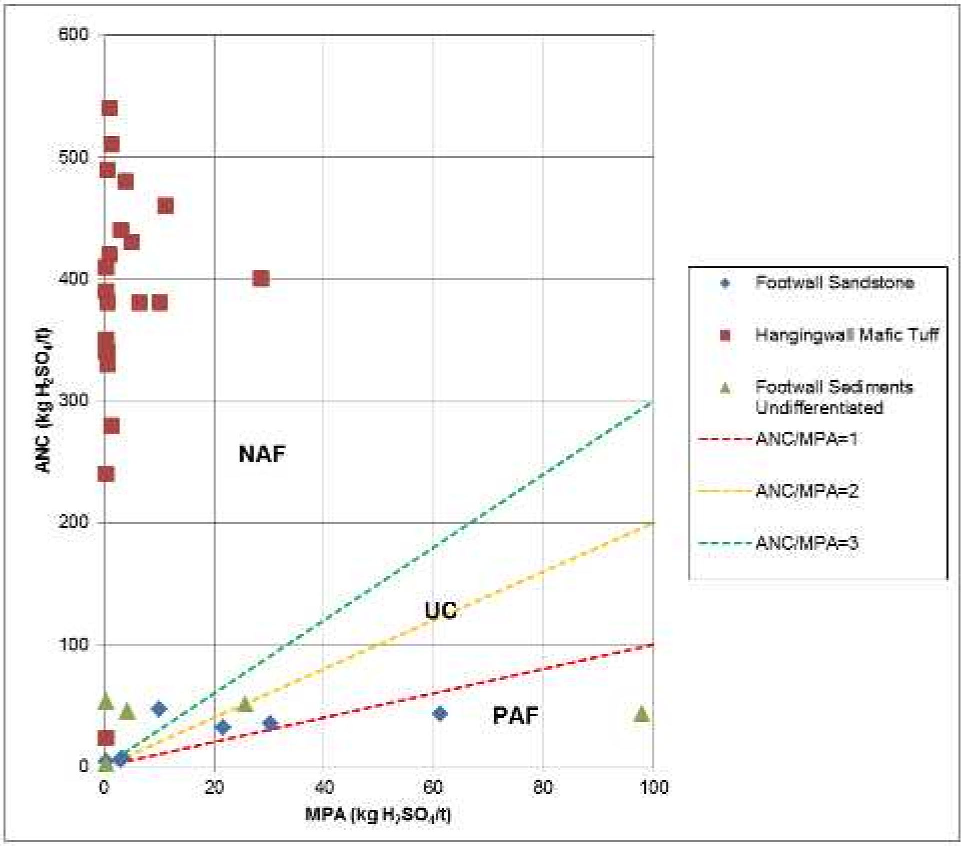

Figure 20-18: Geochemical classification of waste rock samples using NPR

Note: The plot includes lines to show where the NPR (ANC/MPA) values equal 1 and 3, indicating boundaries between PAF, UC and NAF regions on the plot. A line to show where the ANC/MPA value equals 2 is also shown, as values greater than 2 are considered to be unlikely to be problematic with respect to AMD (DITR, 2007).

Table 20-3: Summary of Available Geochemical Characterisation of the Deposit

Parameter	Waste Rock	Mineralization	Comment
Paste pH	7.6-9.4	6-9.1	
Paste EC, µS/cm	38-1800	34-490	
Total sulphur,%	<0.01-2.1% <0.01-0.11% (weathered zone) <0.01-2.1% (fresh zone)	<0.01-3.2 <0.01-0.51% (weathered zone) 0.38-3.2% (fresh zone)	Where sulphur speciation testwork was available (fresh zone waste rock, and ore), sulphide was the dominant form of sulphur, accounting for between 94-99% of the total sulphur content.
Acid neutralising capacity, kgH2SO4 /t	280- 540 (Hanging wall mafic tuff) 1.9-6.7 (other lithologies in the weathered zone) 25-53 (other lithologies in the fresh zone)	<0.5-380 <0.5-160 (sediment -hosted ore)	The readily available ANC[4] of the Hanging wall Mafic Tuff from the weathered and fresh zones were assessed to range up to 50 kgH2SO4/t and 20 kgH2SO4/t, respectively.
Net acid producing potential (NAPP), kgH2SO4/t	-539 to -23 (Hanging wall mafic tuff) Close to zero (Footwall sediment and sandstone)	-350 to -1 (most samples) 8.4 (Graphitic Sediment Hosted Massive Quartz sample)	
Net acid generation (NAG) testing NAG pH NAG acidity, kgH2SO4/t	2.5-9.3 <0.5 - 36	7.9-10.4 <0.5	5 of 6 waste rock samples and 9 mineralization samples generated no acidity. One waste rock sample (Footwall Sediment) was acid generating (NAG pH 2.5; NAG acidity, 36 kgH2SO4/t)
Geochemical Classification NPR AMIRA	NAF (26), UC (5), PAF (2) NAF (4), UC(NAF) (1), PAF (1 - footwall sediments)	NAF (7), UC (4), PAF (1) NAF (9), UC(NAF) 1	

[4] Readily available ANC was assessed using a modified ANC test – details not of testwork method not specified.

20.4 Mineralized Waste Management

It is understood that there is an existing stockpile comprising approximately 300,000 m^3 oxidised, non-mineralized, and waste rock. Any future project is likely to generate additional stockpiles that would need to be managed.

Geochemical characterisation datasets are limited. There is insufficient information to determine the potential for acid, metalliferous drainage from the wastes with any degree of confidence. While oxidised material is currently considered to be non-acid forming (NAF); further data are required to verify that this would be true of oxidised volumes to be mined as part of current plans. Even if classed as NAF, the potential for leaching of elements such as arsenic, antimony and selenium under pH neutral, oxidising conditions should be evaluated.

Some important issues with respect to waste rock management requirements are:

Determination of the volumes of NAF and PAF-classed materials that would require management on the surface.

For high sulphide wastes (whether PAF or NAF-classed), waste storage facilities should be designed so that potential base seepage and surface runoff are managed . This would mitigate the potential for water quality impacts from acidic, metalliferous drainage (PAF wastes) or neutral saline drainage (NAF-classed sulfidic wastes).

Should material be identified that reacts at a very high rate (e.g. high sulphide content material), then it may be advisable to use dump designs and construction methodologies that control and minimise oxidation rates. The objective would be to minimise accumulation of reaction products during storage on the surface. Such products could dissolve in contact waters when stored material is returned underground and is inundated as groundwater rebounds. Minimisation of accumulation rates on the surface will minimise potential impacts on groundwater quality later.

21 Capital and Operating Costs

21.1 Summary

The costs for the project, described in the following section have been derived from a variety of sources, including data provided by Newmarket Gold based on its existing Northern territory operations and SRK' s internal data base.

The open pit mining operating costs have been developed from, unit costs provided by Newmarket Gold, sourced from local contractors and include allowances for Mining, Drill and Blast, grade control, Indirects and road maintenance costs.

The underground mining operating costs are based on unit rates from the existing contracts for Newmarket Gold' s Northern Territory operations and estimates by SRK.

The key drivers are development AUD/m rates, drill and blast AUD/m rates, ground support costs, truck haulage AUD/tkm rates, backfill rates for cemented and waste rock fill. Assumptions have been made for supervision, underground services and mining G&A costs sourced from SRK' s internal database.

The basis of the capital costs estimates for the Process Plant and Infrastructure are presented in Section 17 and described in further detail in Section 21.2.

The mining capital cost estimates are predominantly developed of the operating costs input assumptions described in Section 21.

All cost estimates are provided in 2015 Australian dollars. Escalation, taxes, import duties and custom fees have been excluded from the cost estimates.

The capital and operating cost estimates have been consolidated into the following files:

CGC001_Maud Creek TEM_Stand -alone_Rev5.xlsx;

CGC001_Maud Creek TEM_UR_Rev5.xlsx; and

Maud Creek TEM_ Option Summary_Rev5.xlsx.

21.1.1 Capital cost Summary

Table 21-1 summarises the total capital cost estimate for the Maud Creek Gold Project.

Table 21-1: Maud Creek Gold Project – Capital Cost Estimate

Item	Total (AUD M)
Process Plant	24.9
Surface Infrastructure	2.6
Surface Water Management	7.3
Access Road Upgrade	2.2
Open pit Mobilisation	2.0
Total-Plant and Equipment	**39.1**
Capital Development	16.4
Total Capital Cost	**55.5**

21.1.2 Operating cost Summary

The operating cost estimates applied in this Technical Report are summarised in Table 21-3 and described further in the following sections.

Table 21-2: Maud Creek Gold Project - Operating Cost Summary

	Cost (AUD M)	Cost (AUD/t)	Cost (AUD/oz rec)
Open pit Mining	29.1	5 /t mined	328
Underground Mining	167.6	55 /t mined	342
Processing	125.1	32 /t milled	255
Concentrate transport	66.7	234 /t con	136
Site G&A, Indirects	19.5	5 /t milled	40
Total Operating cost	**408.1**	**105 /t milled**	**1,101**

21.2 Processing Plant Capital Cost

The Processing Plant capital cost estimate for this PEA has been generated from designs and flowsheets described in Section 17 to a preliminary level of accuracy, i.e. +/-30%.

A standard methodology was used for estimating the preliminary capital costs. Processing flowsheets for a stand-alone plant were developed, the process design criteria generated and the flowsheet modelled using the SysCAD process simulation software. The model incorporates equipment sizing calculations developed by Simulus Engineers. These equipment sizes generate a mechanical equipment list which is then costed using a database of recent equipment costs. Additional budget pricing will be obtained from vendors at the next stage of the study. The following process areas listed below were then removed from the capital cost estimate for the Union Reefs treatment option as they exist under the new base case:

- Crushing
- Milling (minor allowance only for hopper and pumping)
- Gravity concentration & gold room
- Tailings thickening
- Tailings storage
- Some reagents
- Raw water
- Water treatment
- Potable water
- Gland water
- Fire water
- Instrument air.

Costs for the other major engineering disciplines such as earthworks, civil, structural, piping, instruments, electrical, site buildings, first fills, laboratory and IT and communications were based on typical industry factors against the mechanical equipment cost. Other factored direct capital cost items include first fills, critical spares and warehouse inventory, laboratory equipment and supplies. All factored costs were adjusted to account for the existing site, systems, cleared areas, etc. Factors were also used for freight, commissioning and vendor support, temporary facilities, mobilisation, demobilisation and EPCM. Other factored items already existing at Union Reefs such as site buildings, IT, communications and network equipment have been completely removed from the cost estimate. Finally, a contingency of 10% was applied due to the preliminary nature of the study to account for other costs not considered.

The capital cost estimate will be refined as part of future study work, with a focus on equipment sizing and costing and integration into the existing processing Union Reefs facility. It is likely there is still some double up of costs because of the preliminary level of the assessment and that the estimate can be reduced further. Examples of potential savings include the assumed requirement for a full separate process water system and full LP and plant air system. Some of the factors used for non-mechanical disciplines are also likely to be lower for a Brownfield site. They have been adjust down but could come down further with a more detailed assessment. Inevitably, other minor modification costs will creep into a Brownfield Project.

At this stage of costing, the equipment sizing is automatically calculated and pricing automatically generated from the process simulation model outputs. Sizing is at a preliminary level of accuracy only. A modular approach will be taken to the design and construction of the processing facility to reduce capital cost, schedule and site construction time.

21.2.1 Infrastructure

The capital cost estimate for the non-mining infrastructure has been generated at a preliminary level of accuracy only. The haul road upgrade costs from the Maud Creek mine site to the Stuart Highway are based on typical unit rates for widening and upgrading the existing access road. This was then benchmarked against SRK database costs for the actual construction costs of a new road for a Northern Territory Project based on first principals. The cost of the 10.7km haul road to Ross Road is AUD2.2M. No costs have been allowed for any potential upgrade of the Ross Road and Que Ping Road intersections with the Stuart Highway. These intersections will be assessed further should the Project proceed to the next stage. An additional 20% contingency has been applied to the road upgrade due to the preliminary level of engineering used in the estimate. These infrastructure capital costs are shown below in Table 21-3.

It is assumed that no additional site buildings are required at Union Reef. It is also assumed that an accommodation village is not required, with the mine workforce instead using accommodation in Katherine and other regional towns. The additional Processing Plant numbers will be accommodated at the Cosmo Village or residentially. Rail, airport and other facilities are available at Katherine. Hydrology, bore field and tailings storage facility costs are discussed separately. Power is already available for processing at Union Reefs, mine site power will be provided by IPP diesel generators sets at negligible cost to Newmarket Gold.

Table 21-3: Infrastructure Capital Cost Estimate Summary

Key Infrastructure	Factor (%)	Total cost (AUDM)
Road Upgrades		2.2
Sub total		2,.2
Contingency (20%)	20	0.44
Grand total		2.64

21.2.2 Sustaining Capital

In addition to the capital costs, economic modelling would normally incorporate a sustaining capital cost for the Processing Plant. This has been removed at this stage of study because processing is through the existing Union Reefs Plant, which has its own sustaining capital allowance. SRK notes that a small allowance may need to be included at the next phase of the Study for any early modifications, debottlenecking and other requirements as the project is ramped up.

This sustaining capital cost excludes future tailings dam lifts. It is understood that there is up to 20 years of existing tailings storage capacity at Union Reefs, another significant benefit. No mining development/sustaining capital have been included as mining is undertaken by contractors.

21.2.3 Duties, taxes and insurances

Goods and Services Tax (GST) and insurance has been excluded from the estimate. The Processing Plant insurance cost is already paid by Union Reefs although there may be a small premium increase to cover the new area of the plant. Mining insurance has been excluded at this stage of the Study. Construction insurance for the plant and infrastructure modifications has not been captured in the capital cost estimate.

21.2.4 Qualifications and assumptions

The capital costs are based on the assumptions above. They will be finalised with award of the work and any refinement of scope.

21.2.5 Exclusions

The following items are specifically excluded from the capital cost estimate:

- GST
- Exchange rate variations
- Additional environmental requirements beyond the scope
- Land acquisition costs
- Cost of handling and disposal of contaminated products
- Escalation
- Owners costs
- Project holding costs.

21.3 Processing Operating Costs

The Maud Creek Gold Project operating costs have been produced for the key cost centres of geology, mining, processing, concentrate transport and general and administrative (G&A) costs. The geology and mining costs were generated by SRK and consolidated into a single 'Mining' cost. The processing and G&A costs were built up by Simulus from first principles. The concentrate transport costs were provided by Bertling and Northline.

This section summarises the operating cost estimates and the basis for them. These costs apply to this PEA. Whilst they have a Project specific basis, they are at a preliminary level of costing only and should be considered to have an accuracy in the order of ±30%. This will be refined should the Project progress to the next phase of study. The operating cost estimates applied in this Technical Report are summarised in Table 21-2 and Table 21-3 and described further in the following sections.

Table 21-4: Maud Creek Gold Project - Operating Cost Summary

	Cost (AUDM)	Cost (AUD/t milled)
Processing (5% contingency)	8.74	18.13
Processing General Administration (5% contingency)	0.18	0.36
ROM Haulage to Union Reef	7.20	14.40
Haul Road Maintenance	0.32	0.64
Concentrate Transport	8.55	17.11
Total	**29.32**	**58.65**

21.3.1 Processing operating cost

Process operating costs have been developed from first principles using the Maud Creek stand-alone facility operating cost model. This was then modified to remove any double up in costs already incurred at Union Reefs. Reagent consumption and grinding media rates are taken from the mass balance outputs and testwork. Unit costs are based on supplier quotes provided to Simulus Engineers by vendors during the previous 6 months (not current contracts at Union Reefs which will be updated at the next phase of study). Labour requirements have been estimated based on the existing Union Reefs organisational structure assuming an extra shift of operators and maintenance is required as well as an extra flotation team member per shift. It uses recent industry assessment for salaries but will be updated at the next phase of study to use the actual salary structure at Union Reefs. Power consumption has been built up from the equipment list and power price based on the local grid prices. The bulk of the fixed costs have been removed as they are largely accounted for already through the Union Reefs fixed operating costs. The breakdown of the processing and G&A costs are provided below in Table 21-5.

The processing operating costs, including maintenance costs are estimated to be AUD18.13/t. This includes a 5% contingency. They were calculated on a throughput of 500 ktpa but through an ongoing Union Reefs Processing Plant already processing ore at an average cost of AUD28.54/t in 2015 for a throughput of 725,002 tonnes. A future review may choose to share the fixed cost savings benefits between the Cosmo underground ores and Maud Creek mineralization.

Power was based on the mechanical equipment list, loads and utilisation factors and network and electricity supply tariffs have been provided at an average unit power cost of AUD0.2177/kWh provided by PWC. The actual electrical power unit cost at Union Reefs will be updated during the next phase of study.

Labour costs are assumed to be largely absorbed by the existing Union Reefs staffing, with one additional flotation technician required for the new flotation circuit for each shift. Currently the Union Reefs Mill operates on a 9 days on 5 days off roster so another shift of operators and shift maintenance has been allowed for a 24/7 operation, an overall increase of 11 people. This also supports the concentrate bagging function. The professional and administrative roles are already filled by Union Reefs' employees and no additional allowance has been made. It remains a lean operating structure. Salaries and wages for the additional personnel are based on a McDonald Gold & General Mining Industries Remuneration Report (Australia). It used the 25th percentile salaries based on the current depressed Australian resources sector. It does not use the existing salary structure at union Reefs. This can be updated at the next stage of Study. Mining and geology manning are addressed in previous sections.

Maintenance and consumable costs including wear liners were based on typical costs from other small scale flotation circuits but applied pro-rata based on the percentage of throughput through the Union Reefs Processing Plant.

Accommodation and messing costs are based on the Cosmo Accommodation Village (200 person capacity) and assuming a local contingent to the extra operating positions. It assumes a drive-in/ drive-out operation of the new starters from Darwin or daily commuting from local towns.

A separate operating cost allowance has been made for haul road maintenance of AUD320,000/year (AUD0.64/t) . The main assumptions are provided below.

Treatment rate of 500 ktpa on a 100% fresh, (will be conservative for the oxide and transitional component of the feed);

Labour costs are covered by Union Reefs existing headcount except for one additional operations shift and one additional operator per shift;

Processing costs are from the ROM pad (the ROM bin front end loader fuel is included);

General administration costs are covered within the existing Union Reefs operating budget;

Additional consumables costs are assigned pro-rata for the feed to plant

An additional AUD100,000/year in freight costs is applied;

Power is provided to Union Reefs from the grid at an average unit power cost of AUD0.2177/kWh;

Average power consumption is 38.5 kWh/t ROM feed;

Reagent costs include delivery to site and are based on recent budget quotes from Redox and Simulus database costs;

Reagent consumptions are based on metallurgical testwork and process simulation;

Water is supplied by the existing Union Reefs infrastructure;

The bulk of the plant control samples are tested on site;

A contingency of 5% has also been provided based on the preliminary level accuracy of the study; and

The estimated costs are direct costs only. Financing and depreciation costs, depletion of mineral property, interest and tax, royalties and other such indirect costs are not included.

Table 21-5: Processing and G&A Operating Costs

Processing and G&A Operating costs	**Unit**	**Cost**
Labour (excluding mining and geology)	AUD	1,419,695
Reagents and grinding media	AUD	1,860,138
Power (excluding mining)	AUD	4,190,388
Consumables	AUD	1,161,039
General and Admin	AUD	170,217
Total	AUD	**8,801,477**
Throughput	tpa	500,000
Labour	AUD/t	2.84
Reagents	AUD/t	3.72
Power	AUD/t	8.38
Consumables (including lab)	AUD/t	2.32
General and Administrative (excl. labour)	AUD/t	**0.34**
Total (Processing)	AUD/t	**17.26**

Processing and G&A Operating costs	Unit	Cost
Total (Processing +5% contingency)	**AUD/t**	**18.13**
Total (GA)	AUD/t	**0.34**
Total (Processing and GA)	AUD/t	**17.60**
Contingency (+5%)	AUD/t	0.88
Total Processing & GA including contingency	**AUD/t**	**18.48**
Haul Road Maintenance	**AUD/t**	**0.64**

Union Reef' s processing operating cost for the 2015 calendar year was AUD28.15/t for 725 kt of dry ore milled, but this includes AUD2.98/t for sodium cyanide and AUD1.29/t for liquid oxygen which isn' t required to treat the Maud Creek mineralization. If an assumption is made that the fixed costs are approximately 35 – 40% at Union Reefs and are therefore removed, the additional cost associated with processing of the Maud Creek mineralization correlates closely.

The process operating cost was also compared against a number of other small flotation plants throughout Australia in the SRK database to provide further benchmarking and validation of the costs. They benchmarked well.

21.3.2 General and Administrative

All G&A costs except for the RoM pad loader diesel are assumed to be fixed costs for Union Reefs and are thus not included in the estimate. This generates the G&A cost of AUD0.34/t including contingency.

21.3.3 ROM Haulage to Union Reef

Ore haulage costs have been benchmarked against similar projects and a rate of AUD0.10 /t/km has been used. This also matches the ore haulage cost from the Cosmo Underground Mine to the Union Reefs Processing Plant. This generates the ROM haulage cost of AUD14.40/t. No contingency has been applied as there is confidence in the haulage contractor cost used.

Haul road maintenance costs include the cost of dry grading, washing delineators and signage, minor repairs and re-sheeting of the road every two years. A typical rate of AUD30,000 /km/year has been applied to the 10.7km haul road from the Maud Creek mine site to Ross Road a total of AUD320,000/year. This generates the haul road maintenance cost of AUD0.64/t. No contingency has been applied.

21.3.4 Concentrate Transport

A concentrate transport cost of AUD234/t concentrate has been developed for the study. It allows for road transport of sea containers from Union Reefs to the Darwin Port, port fees, and shipping to China. This translates to AUD17.11/t of feed processed through the plant.

Preliminary concentrate transportation costs are based on an integrated transport solution supplied by a third party freight transport company. It includes all land transport, shipping, handling and storage. The basis of this costing is provided in Section 19.1 Concentrate Transport. It assumes 36,500 tpa of dry concentrate is transported at 10% moisture in 1.5 tonne bulk bags which are in turn stored in 20' shipping containers.

The cost of purchasing or leasing the 20 foot containers and variable surcharges such as the bunker adjustment factor (BAF) are not included in the transport logistics cost but the bulk of the typical fees such as the port service charge, port security fee, bill loading fee, export declaration fee and PRA lodgement are. The sea freight quote does not allow for the Seaway Bill, insurance, duty payable to customs, GST payable to customs, disbursements payable to customs & any additional quarantine surcharges on arrival, an allowance for fuel surcharge fluctuations or currency uplift. Costs were requested from FH Bertling and Northline and a summary of the budget costs are shown in Table 21-6.

It excludes hazardous goods transport and capacity has not been confirmed (not expected to be an issue at the proposed tonnages).

Table 21-6: Market Costs for Concentrate Transport

	Units	Northline	FH Bertling
Transport from site to Darwin Port			
Cost per container	AUD/container	1,752	1,685
Cost per ton	AUD/t	87.6	84.3
Cost per dry ton	AUD/dmt	97	94
Transport from Darwin to Dalian			
Cost per container	AUD/container	2,838	2,529
Cost per ton	AUD/t	142	126
Cost per dry ton	AUD/dmt	158	140
Transport from Site to Dalian	**AUD/dmt (conc.)**	**255**	**234**
Transport from Site to Dalian	**AUD/dmt (feed)**	**18.62**	**17.11**

The concentrate charges for transport from the Union Reefs Processing Plant to Darwin Port using a road transport has been estimated at AUD94 per dry metric tonne (dmt).

The shipping charges for transport from Darwin Port to Dalian Port have been estimated at AUD140 per dry metric tonne (dmt). The price excludes the variable bunker surcharge (BAF) which is applied at the time of shipment.

The flotation gold concentrate is loaded into 1.5 t capacity bulk bags, which are then loaded into shipping containers ready for transportation. The containers are loaded onto a road train that transports the concentrate by road from the Union Reefs Processing Plant to the Darwin Port. Railway transport has not been considered due to the need for a siding, capital cost, the relatively small volume of concentrate, short LoM and proximity and connectivity to Darwin via a major highway.

The average payload of each container is approximately 20 wet tonnes of concentrate. An additional bag should be loadable but this maintains a level of conservatism in the estimates. A summary of the transport logistics assumptions are shown in Table 21-7. They have been used to estimate the cost of transport.

Table 21-7: Concentrate Transport Logistics

	Units	Value
Plant Operation		
Annual production	t/y	500,000
Operating weeks per year	w/y	52
Recovery to concentrate	%	7.3
Concentrate production	t/y	36,500
Concentrate moisture content	%	10
Container Transport		

	Units	Value
Product per container (wet)	t	20
Product per container (dry)	dmt	18
Containers per week	containers/w	48
Containers per year	containers/w	2,028

The charge for the integrated concentrate transport solution has been estimated at AUD234 per dry metric tonne (dmt). This value correlates well when benchmarked against other small Australian concentrate producers exporting containerised product to China.

At Darwin Port, the containers are stacked until required for ship loading with fortnightly sailings scheduled. Preliminary costs assume Dalian Port as the destination port in China and have a transit time of 29-36 days via Shanghai or Kaoshiung.

The cost of transporting the gold concentrate is included in the G&A cost of AUD0.34/t including contingency.

At the next level of design, logistic cost details would be further developed. The study would investigate whether individual components of the logistics chain could be separated in order to improve the overall cost. It is also possible that with detailed negotiations between the owner and logistics companies may also reduce costs once production was imminent. The next phase of study will also look at whether an additional one or two bags of concentrate can be loaded in each container to further reduce the cost per tonne.

21.4 Concentrate Selling Expenses

The concentrate selling costs are based on indicative terms for the China direct smelting option which makes up the bulk of the gold ounces produced. A portion of the gold is also produced as a gravity gold concentrate and will likely be intensive leached, electrowon and smelted into doré on site at Union Reefs using the existing equipment. Consideration will need to be made in whether the gravity gold can be intensively leached or whether it will need to be upgraded using shaking tables before directing smelting or whether it will require specialised smelting and refining at an established Australian gold refiner.

For financial modelling purposes, the following terms were used as provided by Shangdong Zhong Guo (China Gold Shandong) for processing at their Yantai Gold Smelter.

Concentrate (Direct smelt in China)

35,500 tpa concentrate (7.3% mass pull x 500,000 tpa)

45 g/t gold

Approximately 52,808 oz/year

Payment for 95% of contained gold in the concentrate

USD10/t processing / refining charge

Gravity concentrate into bullion (AGR or equivalent in Australia)

14,802 oz/year

AUD1 oz refinery charge

AUD5k/month for bullion transfer

Both gravity and flotation concentrate terms will be developed further at the next phase of the study.

22 Economic Analysis

22.1 Introduction

The Maud Creek Gold Project financial model was developed by SRK based on the production schedule and assumptions described in the earlier sections. All costs are constant in 2015 AUD with no provision for inflation or escalation.

The annual cash flow projections were estimated over the project life based on capital expenditures, operating costs and revenue. The financial indicators examined included after-tax cash flow, net present value (NPV), internal rate of return (IRR).

It is important to note that the PEA is preliminary in nature that it includes Inferred mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and there is certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have documented economic viability.

22.2 Principal Assumptions

The principal assumptions in the mining schedule described in Section 16 of this Technical Report and the economic parameters in Section 21.

The key project economic assumptions used in preparation of the cash flow analysis have been listed in Table 22-1. SRK notes that the time of reporting the gold price was over AUD1,600 /oz.

SRK has applied a discount rate of 5% on the basis that Newmarket Gold is a Canadian based company and will source funding from US/ Canadian markets. An increase in the discount rate to 8% reduces the After Tax NPV by AUD20M.

Table 22-1: Economic Assumptions

Description	Units	Quantity
Gold Price	Gold AUD/oz	1,550
Exchange Rate	AUD:USD	0.77
Gold Price	USD/oz	1,200
Discount Rate	%	5

22.2.1 Revenue – Sales Price Assumptions

The economic model assumes that concentrate shipments are made at the end of each time period, monthly. The payables of the shipments and associated selling expenses are assumed to occur at these same time periods within the economic model.

The payable metal terms adopted in the economic model are consistent with the current sales contract terms for the gold concentrate grades and quality. It is also assumed that all gold recovered reports to the concentrate.

22.2.2 Taxes

The TEM includes Australian Government taxes of 30% and NT tax of 20%.

All tenements within the Northern Territory, Australia are subject to a Northern Territory Government Minerals Royalty in accordance with the Northern Territory Mineral Royalty Act 1982 (as amended). This royalty is calculated as 20% of the "Net Value" of mine production, where "Net Value" equals the gross revenue from the relevant production unit less the operating costs of the production unit for the year, a capital allowance on eligible capital assets expenditure, eligible exploration expenditure and additional deductions as approved by the Northern Territory Minister for Mines.

The economic analysis treats the Northern Territory Royalty as a Post-Tax cost, as is it is influenced by losses carried forward. SRK notes that Newmarket Gold NT Holdings Pty Ltd carries tax losses that have not been included in the economic analysis. Based on the integration of the Project this has the potential to further impact the value of the Project. With no "losses" carried forward into the Northern Territory Royalty calculation, the estimates value if $107 per ounce. The estimated available tax losses as at 31 December 2015, provided by Newmarket Gold is:

Income tax non-capital losses AUD229.8M

NT Royalty Tax Net Negative Value AUD151.0M.

22.2.3 Royalties

A summary of the Royalty payments is presented in Table 22-2.

Table 22-2: Maud Creek Project Royalties

Project	Parties Involved		Royalty Commitment	Tenements	Comments
Maud Creek	Newmarket Gold	Harmony Gold	1% NSR on Gold	MLN 1978 MCN' s 4145 - 4146 MCN' s 4149 - 4152 MCN' s 4343 - 4348 MCN' s 3839 - 3844	Maud Creek after 250,000 ounces produced
	Newmarket Gold	Mt Carrington Mines	AUD5/ounce gold produced	MCN' s 4218 - 4225	Area south of Maud Creek
	Newmarket Gold	Biddlecombe	1% NSR on all ore/metals	MCN' s 4218 - 4225 Part of EL25054	Area south of Maud Creek

A Northern Territory Build Levy of 0.1% of the cost of construction work for civil structures that form part of the land. For the purposes of this assessment this is assumed to include the process plant, surface infrastructure, tailing storage and the paste fill plant.

22.2.4 Reclamation

Possible salvage value on plant and equipment or profits from the sale of assets has not been included in the economic model. For the purposes of this assessment it has been assumed that cash flow and existing rehabilitation bonds will be used to pay for mine closure as well as any additional reclamation required. It is recommended that this assumption is revised if the study progresses to the next phase.

22.3 Economic Summary

The annual production schedule and cashflow forecasts are presented in Table 22-3. A summary of the project economics are presented in Table 22-4.

Table 22-3: Annual production and Cashflow Estimates

		Yr. 1	Yr. 2	Yr. 3	Yr. 4	Yr. 5	Yr. 6	Yr. 7	Yr. 8	Yr. 9
Tonnes Milled	000 t	345	502	452	476	484	489	385	283	315
Head Grade	g/t	4.6	4.7	4.9	4.2	3.8	4.1	3.8	3.4	3.5
Contained Metal	koz	51	77	72	64	60	65	47	31	36
Capital Expenditure	AUDM	44	7	4	1	0	0	0	0	0
Operating Expenditure	AUDM	43	51	45	47	49	48	40	33	33
Net Revenue	AUDM	67	107	100	90	82	90	65	43	50
Cashflow (After Tax)	AUDM	-26	49	52	42	33	42	24	11	16

Table 22-4: Summary of PEA Results

Parameter/ Result	Units	Quantity
Gold Price	AUD/oz	1,550
Exchange Rate	AUD:USD	0.77
Gold Price	USD/oz	1,200
Mine Life	Years	9.5
Mineral Inventory	‘000 t	3,911
Diluted Gold Grade	g/t	4.2
Contained Gold	koz	528
Gold Recovery (oxide/transitional)	%	85
Gold Recovery (sulphide)	%	95
LOM Recovered Gold	koz	496
Production Rate	ktpa	500
Average Annual Gold Production	koz	52
Peak Annual Gold Production	koz	70
Annual Tonnes Concentrate (Dry)	kt	30
Concentrate Grade	g/t con	45
LOM Operating Cost	AUDM	408
LOM Cash Operating Cost	AUD/oz	1,101
Total Operating Costs/tonne Milled	AUD/t	105
Net Revenue (less selling expenses)	AUDM	725
Pre-Production Capital cost	AUDM	42
Sustaining Capital Cost (LOM)	AUDM	14

Table 22-5: Project Economics

	Units	Union Reefs Plant Quantity
Mining		
Mine Life	years	9.5
Mineral Inventory	'000 t	3,911
Gold Grade	g/t	4.2
Contained Gold	koz	528
Open Pit Tonnes Ore	kt	634
Open Pit Gold Grade	g/t	5.12
Open pit Contained ounces	koz	104
Total Waste Mined	'000 t	5,000
Strip Ratio (O/P)	t:t	8
Underground Tonnes Ore	'000 t	3,276
Underground Gold Grade	g/t	4.02
Underground Contained ounces	koz	423
Processing		
LOM Tonnes Milled	'000 t	3,911
Production Rate	ktpa	500
Average Annual Gold production	koz	52
LOM Recovered Gold	koz	496
Metallurgical Recovery		
Transitional to Float	%	65
Transitional to Gravity	%	20
Fresh to Float	%	75
Fresh to Gravity	%	20
Annual Tonnes Concentrate (Dry)	kt	30
LOM Tonnes concentrate (Dry)	kt	285
Concentrate Grade	g/t con	45
Economics		
Gold Price	AUD/oz	1,550
Exchange Rate	AUD:USD	0.77
Gold Price	USD/oz	1,200
LOM operating cost	AUDM	408
LOM operating cost	AUD/t milled	104
LOM operating cost (Payable)	AUD/oz	856
LOM operating cost (Payable)	USD/oz	662
LOM Payable (Saleable Gold)	koz	477
Net Revenue (less selling expenses)	AUDM	725
Capital cost	AUD M	56
Pre-tax Cashflow	AUDM	261
Pre-tax NPV (5)	AUDM	201
Pre-tax IRR	%	116
Payback Period (pre-tax)	Qtr	6

	Units	Union Reefs Plant Quantity
After Tax Cash Flow	AUDM	182
After Tax NPV (5)	AUDM	137
After Tax IRR	%	80
Payback Period (After Tax)	Qtr	6

22.3.1 Sensitivity

Cash flow sensitivities (+/- 15% and 30%) to gold price, exchange rate (AUD:USD), metallurgical gold recovery, mill feed gold grade, capital costs and operating costs has been completed and presented in Figure 22-1.

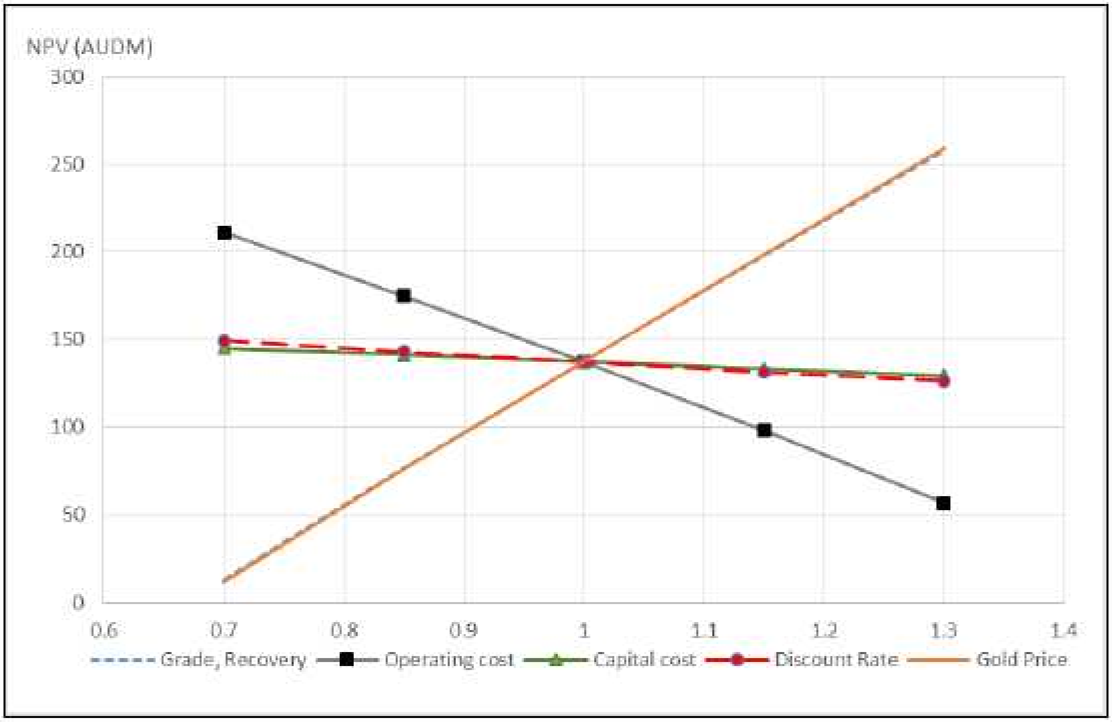

Figure 22-1: Sensitivity Analysis

The key drivers are the revenue assumptions (Grade, Recovery and Gold Price) and the operating cost assumptions. Sensitivity to the Gold Price is presented in Table 22-6.

Table 22-6: Gold Price Sensitivity

Gold Price (AUD/oz)	1,400	1,450	1,500	1,550	1,600	1,650	1,700
Pre-Tax NPV5% (AUDM)	145	163	182	201	220	239	257
Pre-Tax IRR%	85	95	106	116	127	138	150
After-Tax NPV5% (AUDM)	98	111	124	137	150	163	177
After-Tax IRR%	59	66	73	80	87	94	102

SRK notes that Capital Cost estimates used for the sensitivity analysis excludes the capital development as the cost driver for this is the unit operating costs. Impact on the capital development costs is included in the operating cost sensitivity.

A breakdown of the operating cost components, including underground capital development is presented in Figure 22-2.

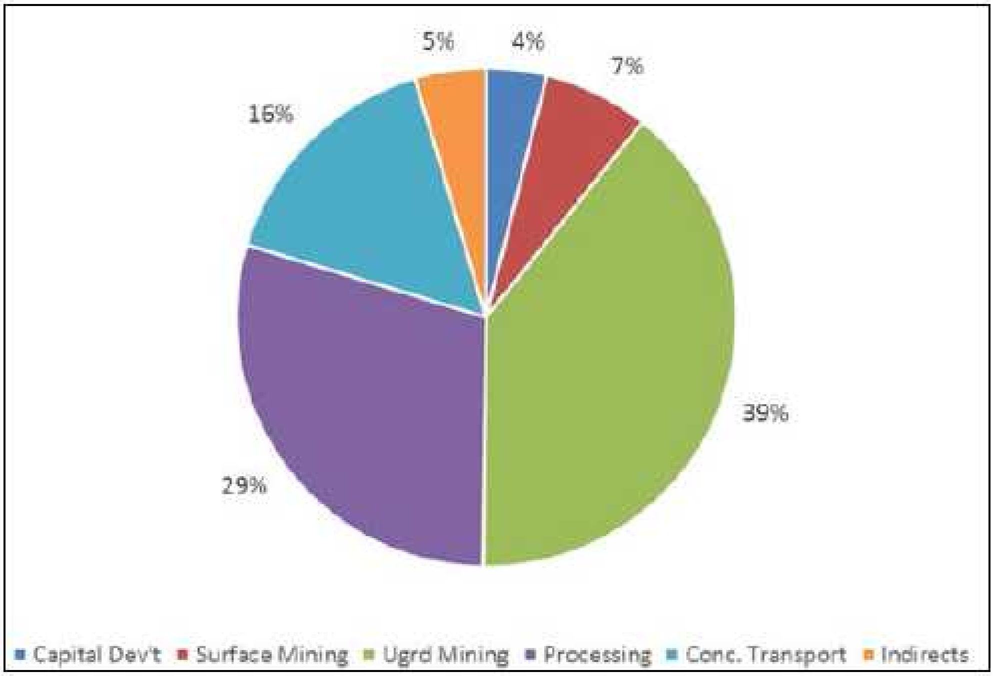

Figure 22-2: Breakdown of Impact of Operating Cost Sensitivity

23 Adjacent Properties

Other projects in the region managed and owned by Newmarket Gold include:

The Pine Creek Gold Project (Figure 23-1), located approximately 190 km south southeast of Darwin accessible by the Stuart Highway, directly to the west of the Township of Pine Creek.

The Union Reefs Gold Project (Figure 23- 2), located approximately 180km south- southeast of Darwin accessible by the Stuart Highway, directly to the north of the Township of Pine Creek.

The Burnside Gold & Base Metals Project (Figure 23-3), located approximately 130 km to the South-southeast of Darwin accessible by the Stuart Highway.

The Moline Gold and Base Metals Project (Figure 23-4), located approximately 40 km to the Northeast of the Township of Pine Creek, accessible by the Kakadu Highway.

The Yeuralba Gold and Base Metals Project (Figure 23-5), located approximately 45 km to the northeast

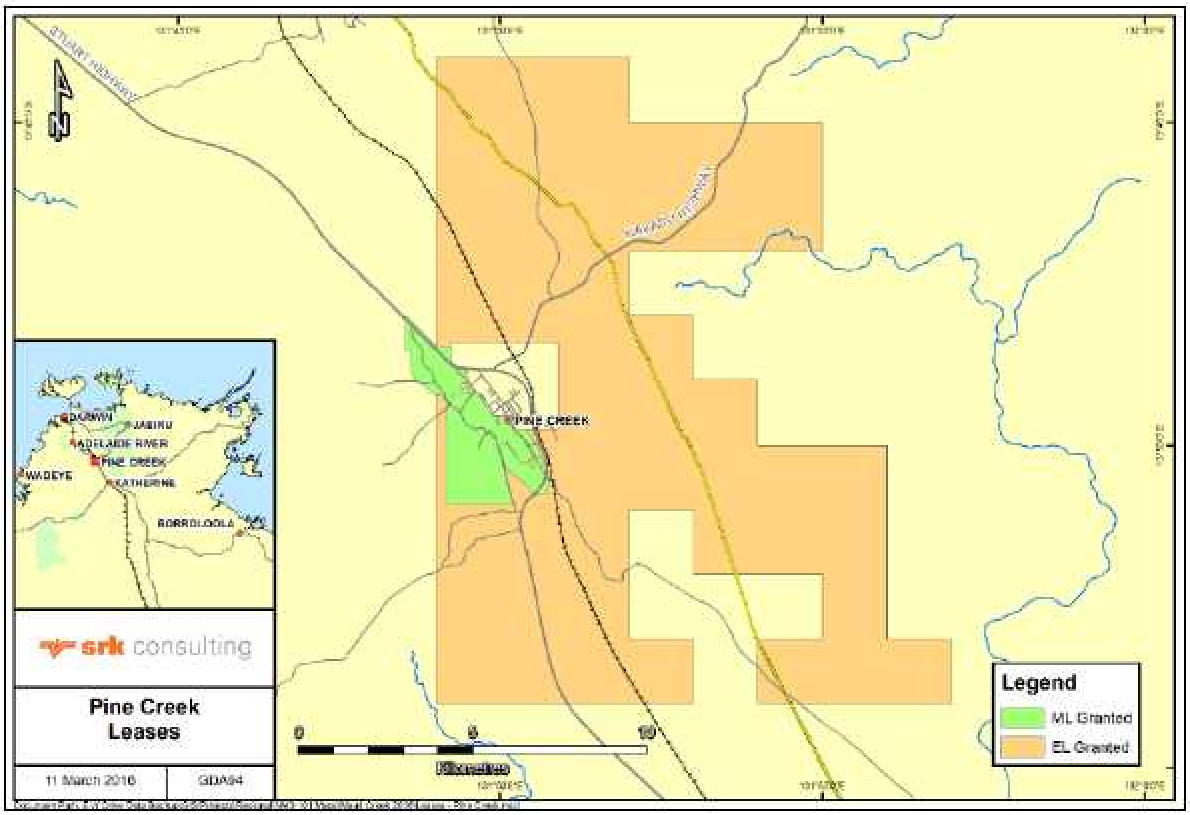

Figure 23-1: Pine Creek Gold Project Tenements

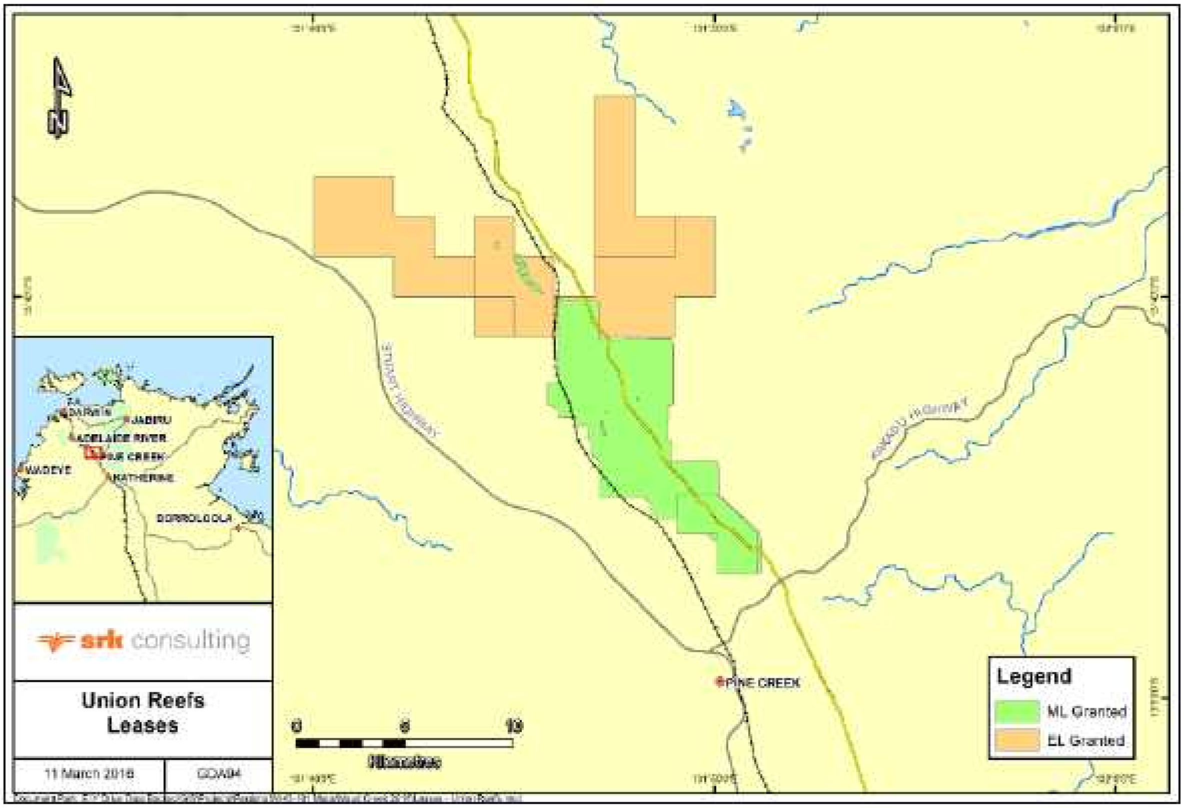

Figure 23-2: Union Reefs Gold Project Tenements

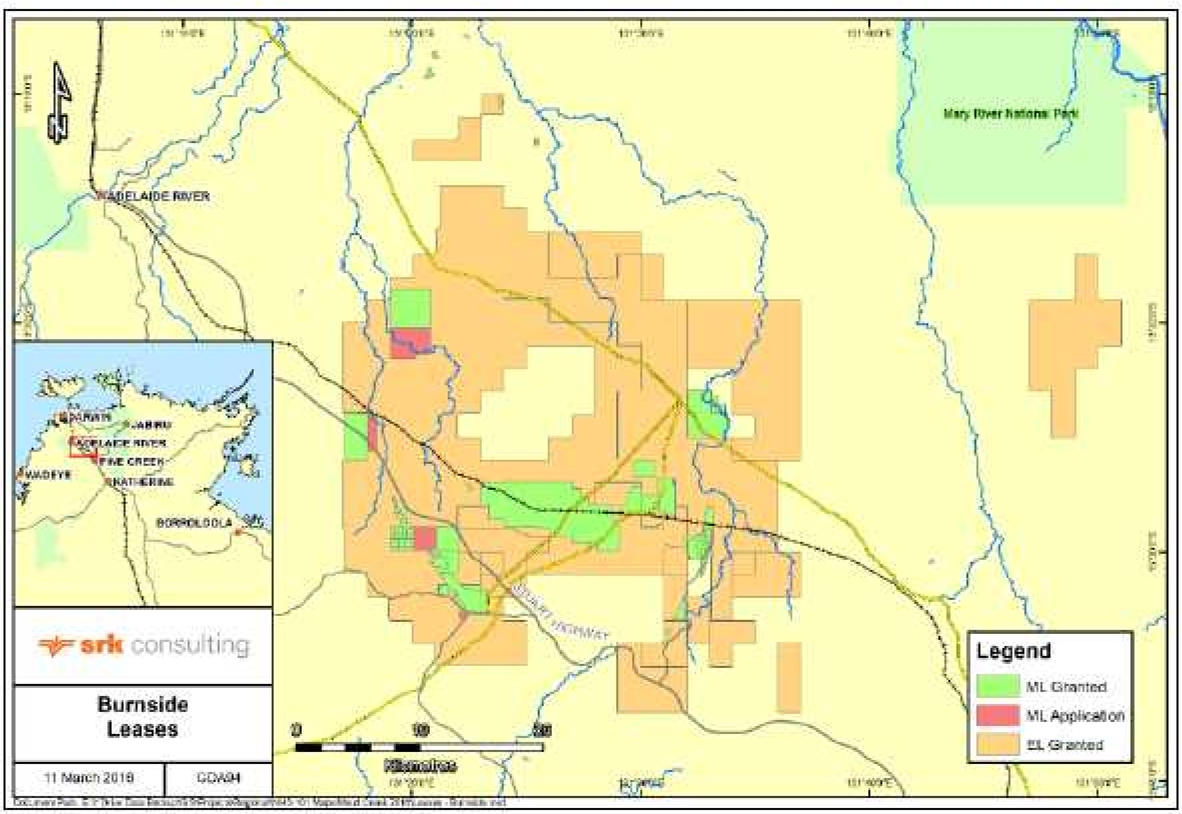

Figure 23-3: Burnside Gold Project Tenements

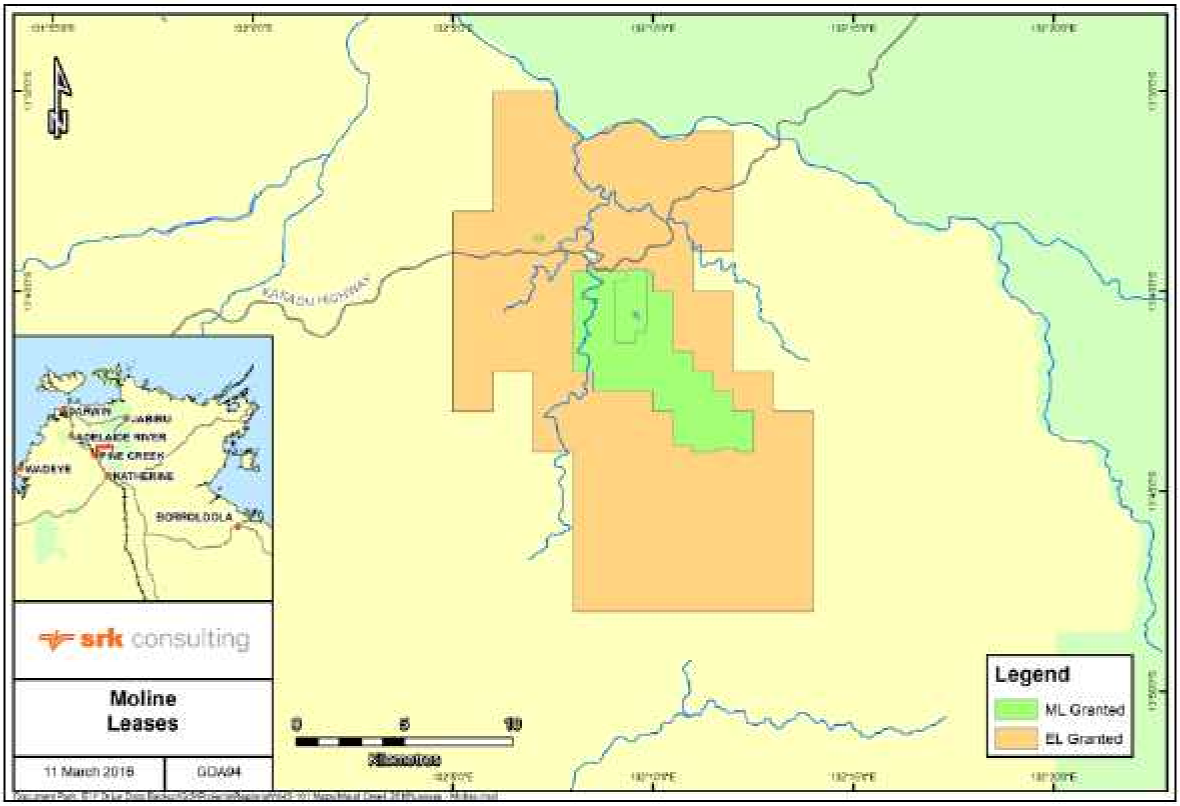

Figure 23-4: Moline Gold Project Tenements

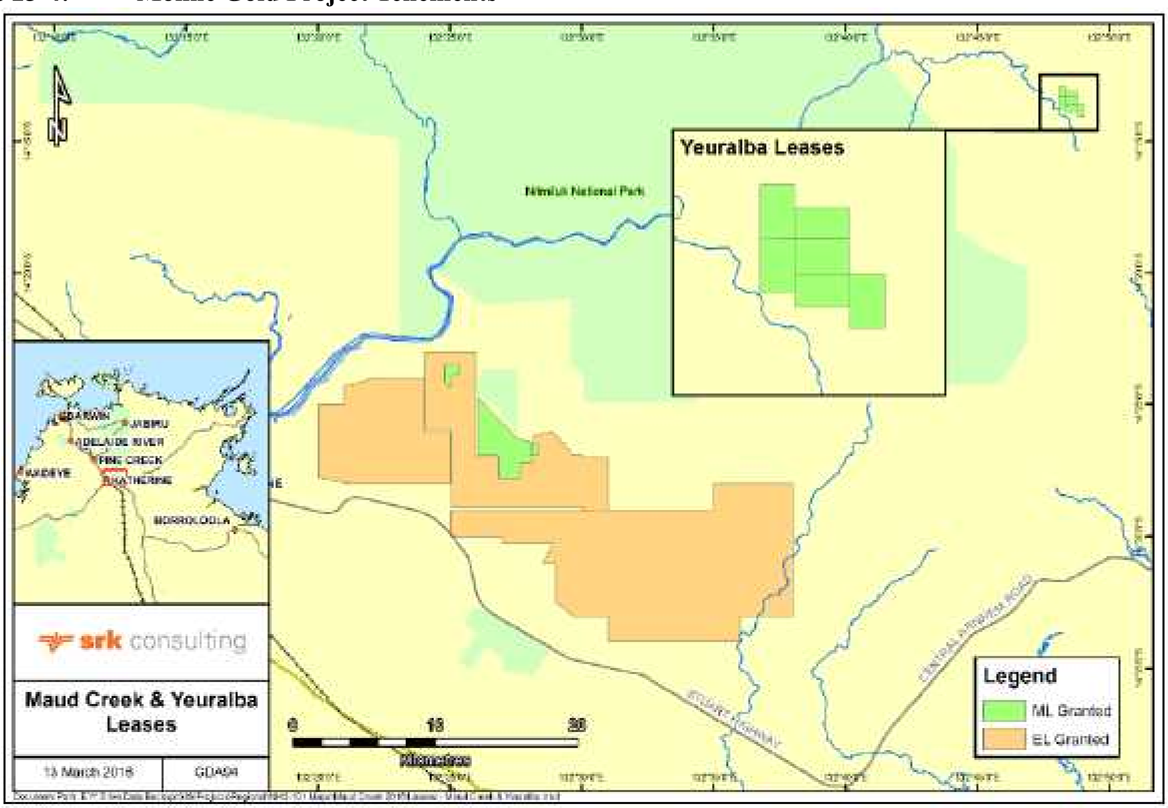

Figure 23-5: Maud Creek & Yeuralba Gold Project Tenements

24 Other Relevant Data and Information

No other relevant / material data has been excluded from the Technical Report.

25 Interpretation and Conclusions

The modelling of vein and grade volumes for the 2015 estimate takes a very different approach to the 2012 model as described in the sections above. The 2015 model incorporates a detailed structural, vein and lithological model in the construction of the various estimation domains. This was a deliberate decision as it was SRK' s understanding from discussions with Newmarket Gold and from reading previous reports that there were potential deficiencies in the very linear grade only approach previously used. Concerns had been expressed in some previous reports that insufficient attention had been paid to the geology and that the previous models may have diluted a high grade, geologically controlled core to the main zone thereby creating a model that underestimated grade and overestimated tonnages at economic cut-offs.

There are a number of differences between the 2012 and 2015 modelling approach. The 2015 model uses the following:

- Pure geology to define the main and minor vein domains. The 2012 used grade only. Consequently the 2015 vein model contains considerably lower tonnage and slightly elevated grade in comparison;

- Grade halos to capture both high and low grade outside the geological veins. This captures low grade material that was not modelled in 2012 which may be of value in an open pit scenario; and

- Orientation controls on the grade halos derived by the combined fault / lithology contact model resulting in multiple orientations and fattening around fault and contact intersections.

The 2015 model is more robust in terms of its geological basis and this has led to a slightly higher grades but a reduction in contained gold. Only further drilling can define true connectivity of the mineralization in widely spaced areas.

A discussion on the risks and opportunities in the Mineral Resource model as discussed in Table 25-1.

Table 25-1: Mineral Resource Model Risks and Opportunities

Project Element	Economic Risk Level	Comment	Opportunity
Database - Exploration data	Low	Historical and recent data have been re-collated and re-validated for this Mineral Resource estimate.	
Assaying	Low	QAQC for recent and older assaying shows no material issues. Arsenic assaying has incomplete coverage.	Additional assaying for Arsenic may be beneficial depending on the processing method.
Surveying	Low	Both collar surveys and downhole surveys completed to a high level of accuracy for recent drilling. Representative collars resurveyed for older drilling with no significant discrepancies.	
Geology	Low	Detailed logging and interpretation together with evidence from both regional structural features and detailed in pit mapping informs the geological understanding	Additional drilling may be able to add detail to the interaction of structures controlling mineralization at depth.

Project Element	Economic Risk Level	Comment	Opportunity
Geological modelling	Low	A detailed structural and lithological model has been built and incorporated into the estimation domain construction.	Additional drilling may be able to add detail to the interaction of structures controlling mineralization at depth.
Resource Estimation	Low	Ordinary kriging cross checked and validated with theoretical grade tonnage curves and alternative search parameters has been used.	The project may benefit from simulation studies or non-linear estimates if detailed studies at selective mining unit block sizes are required in the future.

The absence of suitable data has led to low-confidence in the geotechnical conditions. Additional data will be required to improve confidence and refine decisions on mining methods and the mine design. The mining method studies are linked to the decision on the location of the Processing Plant and the availability of pastefill.

The Maud Creek mineralization has been subject to extensive metallurgical testing. However there remain some gaps that should be covered for future studies that could reduce the future design risks if it was decided to undertake additional testwork earlier. This would include, lithological domain characteristics, SAG milling parameters, testing samples from depth, flotation testwork and concentrate production and specification.

PEA Results

It is important to note that the PEA is preliminary in nature that it includes Inferred mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and there is certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have documented economic viability.

Based on the economic findings of the PEA, SRK concludes that the Project has merit and that Newmarket Gold consider progressing the study to the next level of detail.

In doing so SRK provides recommendations that Newmarket Gold consider progressing the additional work programs (outlined in Section 26) to better understand the key technical risk areas to support future evaluations.

26 Recommendations

Drilling Campaign

Infill drilling in the parts of the Mineral Resource currently classified as Indicated and Inferred would enable an upgrade of the Mineral Resource Classification. An approximate meterage and cost to complete this from surface down to 850 mRL is provided in Table 26-1 assuming the drilling takes place from surface. A number of sections of the geological model remain open down dip with good grades seen in the last hole down dip. Extension drilling is recommended to test these areas. Meters and costs to complete these are shown in Table 26-1 assuming drilling from surface. Costs are based on RC collars and 50m diamond drill tails.

Table 26-1: Recommended Geological drilling

Target	Current exploration status	Potential End of 2016 Status	Description	Drilling (m)
Infill Drilling	Indicated and/or Inferred	Measured and/or Indicated	Increase confidence in estimated Mineral Resource	9,200
Extension drilling	Down dip or along strike from current Mineral Resource	Indicated and/or Inferred	Close off or extend Mineral Resource volumes	2,200

Geotechnical

If an additional phase of geotechnical work is undertaken, SRK recommends that a drilling program be considered to provide additional geotechnical data and infrastructure for hydrogeological testwork. These programs should be combined where possible to assist in any further resource definitions requirements.

Based on further drilling and geotechnical data, further studies should be undertaken to improve the confidence of the understanding of the geotechnical domains and to provide a basis for the improving the design guidelines.

Recommendations regarding phase two work, summarised in Table 26-2, includes the following:

Dedicated geotechnical drilling program to:

- Target area' s that require additional data such as the footwalls sediments, and crown pillar area;
- Drilling proposed locations of LOM infrastructure – portals, declines, vent shafts etc.;
- Improve understanding of structural characteristics including continuity and orientation variability;
- To collect representative samples for laboratory testing;

Mapping and photogrammetry in current pit;

Geotechnical laboratory testing using NATA certified laboratories;

Detailed backfill design for potential mining methods (including CRF);

Potential stress measurement testing (AE or DRA); and

Numerical modelling of proposed mining layout and sequence (including crown pillar and central pillar sequence)

Table 26-2: Geotechnical drill program targets and estimated depths

Description	Drillholes (No.)	Depth (m)	Total (m)
To investigate sediments at depth	4	600	2,400
To supplement data and sediments across mid depths of mine	4	500	2,000
Portal and boxcut investigation drill holes	3	50	150
To investigate Infrastructure located in sediments (can be separated into multiple shorter drill holes)	1	600	600
Open Pit geotechnical holes – will provide data on crown pillar area and upper sediments	6	150	900
Total meters for proposed program			**6,050**

Geometallurgy

Insufficient information was available from the metallurgical reports to create preliminary spatial domains for the physical processing parameters, such as grindability. The metallurgical report Core Process Engineering Report No. 140-001 outlined JK Drop Tests which had been conducted, resulting in a Bond ball Work Index of 18-19 kWh/tonne for the main lode within the deposit. Previous geology reports, as well as a site visit to inspect the core, gave an indication of the mineralogy of the Maud Creek deposit, which has an impact on the hardness of the mineralization. From a geometallurgical perspective, between and within each mineralized domain there will likely be a range of hardness values which will need to be established. Quartz alteration at Maud Creek has been identified as varying between the primary lode (high percentage quartz veining), moderate (footwall and hanging wall lodes stock work veining) to low (sandstone and tuff country rocks). However variations in silica content within these lodes will definitely occur, as alteration boundaries are normally pervasive across lithology boundaries. A possible way to better define the hardness parameters is the use of proxies. If silica analysis is included in any future assay testing of the mineralized zones, this can be used to identify target areas for JK drop weight tests. A regression calculation between the A*b result, Bond work indices and other comminution parameters versus the silica content can then be determined, allowing a predictive model of the hardness to be created. Mineral analysis would need to be conducted to ensure the silica content is reflecting quartz alteration which has a high hardness (Mohs scale 7) and not feldspar (Mohs scale 5-6) or mica minerals (Mohs scale 2-3) which have lower hardness and varying crystallography, resulting in different grinding behaviours. The Geology of the Maud Creek Gold Deposit and Maud Creek Reconciliation report by AngloGold in 2000 identified plagioclase laths and alkali feldspar minerals in the Andesite unit located to the south of the deposit, but not within the mineralized lodes. However, sericite (a type of muscovite) and chlorite alteration (most likely the mineral clinochlore) are both quite prevalent in all three mineralized zones, so their prevalence would need to be established as well. Similarly, the extent of the hematite and marcasite alteration noted in all three mineralized zones (present as pseudomorphs of pyrite in the oxide zone) needs to be quantified, as all three will have moderate-high hardness (6-7), but also a brittle tenacity upon breakage, which will influence the grindability.

The Maud Creek mineralization is going to be very hard and abrasive so finding a correlation with proxies such as silica, for example, is recommended by SRK. Equally, determining correlations between arsenic, sulphur and gold to help generate the flotation relationship correlations, and approximated the amount of arsenic in the final concentrate is also recommended. The arsenic and sulphur contents will also be a good indicator of recovery.

Mining

Should the project proceed SRK recommends that a range of sensitivities/ scenarios are considered at a high level to inform the project team of the implications of these constraints. The sensitivities should include combinations of considering the open pit oxide mineralization as a mill feed source with both the underground preferences retained and removed. Conventional open pit mining techniques are proposed.

The opportunity exists to modify the design to interact with the open pit design but this will potentially have constraints on production continuity.

The mine design and schedule should be revised based on the findings of the geotechnical review.

Processing & Infrastructure

The following additional recommendations are provided based on the findings of this PEA and the base case being to process mineralization at the Union Reefs Processing Plant, after modifications, rather than at a standalone processing facility:

Undertake a further review of the metallurgical testwork to understand the behaviour of the oxide mineralization which were largely excluded from the previous Phase 1 review as processing of oxides was not part of the base case.

Undertake testwork to understand the intensive leach behaviour of the gravity gold concentrate, generate flotation concentrate for customer testing and assessment, other minor testwork as discussed in the main body of this report.

Undertake a specialist comminution circuit capacity study processing Maud Creek mineralization through Union Reefs Processing Plant.

Undertake a more detailed integration study of a new flotation, dewatering and concentrate storage circuits into the Union Reefs Processing Plant in consultation with the Union Reefs and Newmarket Gold’ s technical groups. This would include scope of work, opportunity upgrades at Union Reef, tie-in requirements and the campaign operating philosophy.

Update the capital cost to suit the next level of study with Union Reefs as the base case.

Update the operating costs to suit the next level of study with Union Reefs as the base case, using their power, labour, accommodation and other costs. Agreement on the share of benefits to Maud Creek and Cosmo projects is also required.

Develop cost of upgrading the access road from the Maud Creek Mine Site into Katherine.

Progress discussions with the local Government authorities to confirm in principal that haulage through Katherine and up to Union Reefs is acceptable – this is a potential fatal flaw to the base case processing option.

Confirm capital upgrade requirements attributable to Newmarket Gold (if any) to the main Stuart Hwy.

Confirm road train maximum tonnage acceptable to Local and Territory Government.

Progress flotation concentrate off-take discussions, to provide more confidence in terms and conditions.

27 References

Bremner, P. and Edwards, M. 2012. *Report on the Mineral Resource and Mineral Reserve of the Maud Creek Gold Project*.

Pakalnis, R C. Poulin, R. and Hadjigeorgiou, J. 1995. *Quantifying the cost of dilution in underground mines,* Mining Engineering, 47(12): 1136-1141.

Scoble, M.J., Moss, A., 1994. *Dilution in underground bulk mining: Implications for production management, Mineral Resource evaluation II, methods and case histories*, Geological Society Publication No. 79, pp. 95-108.

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NOTE

The enclosed Technical Report, titled "Amended Technical Report Preliminary Economic Assessment of the Maud Creek Gold Project, Northern Territory, Australia" and dated as of 18 May, 2016 amends the Technical Report "Amended Technical Report Preliminary Economic Assessment of the Maud Creek Gold Project, Northern Territory, Australia" and dated as of 16 May, 2016 filed by Newmarket Gold Inc.

The enclosed Technical Report is being filed to clarify the following information:

1. Correction of the $/recovered ounce LoM Operating Cost in Table ES-1, Table 21.2 and Table 22.4.

   The error was a reporting calculation error and does not affect other reported economic/ financial metrics.

2. Other edits were also done throughout the text, which SRK considers do not constitute a material change.

Amended Technical Report
Preliminary Economic Assessment
of the Maud Creek Gold Project,
Northern Territory, Australia

Prepared for:

Newmarket Gold Inc

According to National Instrument 43-101 and Form 43-101 F1

Prepared by:

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Simon Walsh, BSc (Extractive Metallurgy), MBA Hons, MAusIMM (CP), GAICD, Associate Principal Consultant.
Date of Report: 18 May 2016
Effective Date: 15 April 2016

Date and Signature Page

SRK Project Number: CGC001

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Newmarket Gold Inc
Maud Creek Gold Project
Northern Territory, Australia

Project Manager: **Peter Fairfield**
Date of Report: **18 May 2016**
Effective Date: **15 April 2016**

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Important Notice

This technical report has been prepared as a National Instrument 43-101 Technical Report, as prescribed in Canadian Securities Administrators' National Instrument 43-101, Standards of Disclosure for Mineral Projects (NI 43-101) for Newmarket Gold Inc. (Newmarket Gold). The data, information, estimates, conclusions and recommendations contained herein, as prepared and presented by the Authors, are consistent with:

information available at the time of preparation;

data supplied by outside sources, which has been verified by the authors as applicable; and

the assumptions, conditions and qualifications set forth in this technical report.

CAUTIONARY NOTE WITH RESPECT TO FORWARD LOOKING INFORMATION

This document contains forward-looking information as defined in applicable securities laws. Forward looking information includes, but is not limited to, statements with respect to the potential future production/ mill feed, costs and expenses of the Preliminary Economic Assessment (PEA) and project; the other economic parameters, as set out in this technical report, including; the success and continuation of exploration activities, including drilling; estimates of Mineral Resources; the future price of gold; government regulations and permitting timelines; requirements for additional capital; environmental risks; and general business and economic conditions. Often, but not always, forward-looking information can be identified by the use of words such as plans, expects, is expected, budget, scheduled, estimates, continues, forecasts, projects, predicts, intends, anticipates or believes, or variations of, or the negatives of, such words and phrases, or statements that certain actions, events or results may, could, would, should, might or will be taken, occur or be achieved. Forward-looking information involves known and unknown risks, uncertainties and other factors which may cause the actual results, performance or achievements to be materially different from any of the future results, performance or achievements expressed or implied by the forward-looking information. These risks, uncertainties and other factors include, but are not limited to, the assumptions underlying the production estimates not being realized, decrease of future gold prices, cost of labour, supplies, fuel and equipment rising, the availability of financing on attractive terms, actual results of current exploration, changes in project parameters, exchange rate fluctuations, delays and costs inherent to consulting and accommodating rights of local communities, title risks, regulatory risks and uncertainties with respect to obtaining necessary permits or delays in obtaining same, and other risks involved in the gold production, development and exploration industry, as well as those risk factors discussed in Newmarket Gold' s latest Annual Information Form and its other SEDAR filings from time to time. Forward-looking information is based on a number of assumptions which may prove to be incorrect, including, but not limited to, the availability of financing for Newmarket Gold' s production, development and exploration activities; the timelines for Newmarket Gold' s exploration and development activities on the property; the availability of certain consumables and services; assumptions made in Mineral Resource estimates, including geological interpretation grade, recovery rates, price assumption, and operational costs; and general business and economic conditions. All forward-looking information herein is qualified by this cautionary statement. Accordingly, readers should not place undue reliance on forward-looking information. Newmarket Gold and the authors of this technical report undertake no obligation to update publicly or otherwise revise any forward-looking information whether as a result of new information or future events or otherwise, except as may be required by applicable law.

NON-IFRS MEASURES

This technical report contains certain non-International Financial Reporting Standards measures. Such measures have non standardized meaning under International Financial Reporting Standards and may not be comparable to similar measures used by other issuers.

Table of Qualified Persons and Contributors

Section	Description	Nominated QP	Contributors
	Executive Summary	Peter Fairfield	All
2	Introduction	Peter Fairfield	
3	Reliance on Experts	Peter Fairfield	
4	Property Description and Location	Peter Fairfield	
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Peter Fairfield	
6	History	Danny Kentwell	
7	Geological Setting and Mineralization	Danny Kentwell	
8	Deposit Types	Danny Kentwell	
9	Exploration	Danny Kentwell	
10	Drilling	Danny Kentwell	Kirsty Sheerin
11	Sampling Preparation, Analysis and Security	Danny Kentwell	Kirsty Sheerin
12	Data Verification	Danny Kentwell	Kirsty Sheerin
13	Mineral Processing and Metallurgical Testing	Simon Walsh	
14	Mineral Resource Estimates	Danny Kentwell	
15	Mineral Reserve Estimates	Peter Fairfield	
16	Mining Methods	Peter Fairfield	Anne-Marie Ebbels, Scott McEwing
17	Recovery Methods	Simon Walsh	
18	Project Infrastructure	Simon Walsh	Peter Fairfield
19	Market Studies and Contracts	Peter Fairfield	Simon Walsh
20	Environmental Studies, Permitting and Social, or Community Impact	Peter Fairfield	Lisa Chandler, Ken Redwood
21	Capital and Operating Costs	Peter Fairfield	Simon Walsh
22	Economic Analysis	Peter Fairfield	
23	Adjacent Properties	Peter Fairfield	All
24	Other Relevant Data and Information	Peter Fairfield	All
25	Interpretation and Conclusions	Peter Fairfield	All
26	Recommendations	Peter Fairfield	All
27	References	Peter Fairfield	All

List of Abbreviations

Abbreviation	Meaning
2D	two dimensional
3D	three dimensional
AAS	atomic absorption spectroscopy
ALS	ALS Minerals
AMC	AMC Consultants Pty Ltd
Amdel	Amdel Limited Mineral Services Laboratory
ANFO	ammonium nitrate-fuel oil
ASL	above sea level
ATCF	after tax cash flow
Au	gold
Au	gold equivalent
AUD	Australian dollar
BAppSc	Bachelor of Applied Science
BCom	Bachelor of Commerce
BD	bulk density
BEng	Bachelor of Engineering
BIOX®	Bacterial Oxidation plant BIOX
BSc	Bachelor of Science
Cambrian	Cambrian Mining Limited
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIP	carbon-in-pulp
(CP)	Chartered Professional of The Australasian Institute of Mining
	and Metallurgy
CRF	cemented rock fill
dmt	dry metric tonne
DTM	digital terrain model
EM	electromagnetic
EPA	Environmental Protection Agency
EVC' s	Ecological Vegetation Classes
FAR	fresh air rise
FAusIMM	Fellow of The Australasian Institute of Mining and Metallurgy
GDip	Graduate Diploma
GEF	Gold and Exploration Finance Company of Australia
GMA/WMC	Gold Mines of Australia/Western Mining Corporation
GPS	global positioning system
GST	goods and services tax
g/t	grams per tonne

HBr	hydrobromic acid
HCl	hydrochloric acid
HR	hydraulic radius
Hwy	Highway
ICP - AES	inductively couple plasma atomic emission spectroscopy
ID^2	inverse distance squared
ID^3	inverse distance cubed
IP	induced polarisation

Abbreviation	Meaning
IRR	internal rate of return
JORC	Joint Ore Reserves Committee
kg	kilogram
kL	kilolitre
km	kilometer
koz	kilo ounces
kt	kilotonne
ktpa	kilotonnes per annum
ktpm	kilotonnes per month
kV	kilovolt
kVA	kilovolt ampere
kW	kilowatt
kWh	kilowatt hour
L	litres
LHD	load-haul -dump
LOM/ LoM	life of mine
L/s	litres per second
M	million
Ma	million years
Newmarket Gold	Newmarket Gold Inc
MAusIMM(CP)	Member of The Australasian Institute of Mining and Metallurgy
mg/kg	milligrams per kilogram
mg/L	milligrams per litre
mH	meters high
ML	million litres
mm	millimeters
MMI	mobile metal ion
Moz	million ounces
mRL	meters reduced level
MRSD Act	Mineral Resources (Sustainable Development) Act 1990
Mtpa	million tonnes per annum
m^3	cubic meters
m^3/s	cubic meter per second
$m^3/s/KW$	cubic meter per second per kilowatt
MVA	megawatt ampere
mW	meters wide
MW	megawatt

NI 43-101	National Instrument 43-101
NPV	net present value
OH & S	Occupational Health and Safety
Onsite	Onsite Laboratory Services
ozs	ounces
PEA	Preliminary Economic Assessment
PhD	Doctor of Philosophy
Planet	Planet Resources Group NL
POX	Pressure oxidation

Abbreviation	Meaning
QA/QC	quality assurance/quality control
QP	Qualified Person
RAR	return air raise
RC	reverse circulation
ROM	run-of -mine
SD	standard deviation
SRK	SRK Consulting (Australasia) Pty Ltd
t	tonnes
tpa	tonnes per annum
t/mth	tonnes per month
TSF	tailings storage facility
TSX	Toronto Stock Exchange
UCS	unconfined compressive strength
USD	US dollars
V	volt

Executive Summary

Introduction

SRK Consulting (Australasia) Pty Ltd (SRK) was engaged by Newmarket Gold to undertake a study on the Maud Creek Gold Project (Maud Creek or the Project) and prepare a Technical Report summarising the findings of a Preliminary Economic Assessment (PEA).

In early July 2015, Newmarket Gold Inc. merged with Crocodile Gold Corp. (Crocodile Gold) to form a new Canadian, Toronto Stock Exchange listed gold mining company named Newmarket Gold that has 100% ownership of the Maud Creek Project.

The work and this Technical Report were prepared by SRK following the guidelines of the Canadian Securities Administrators' NI 43-101 and Form 43-101 F1.

The Mineral Resource statement reported herein was prepared in conformity with generally accepted Canadian Institute of Mining, Metallurgy, and Petroleum' s (CIM) Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines.

Scope

The scope of this Technical Report documents the findings of the PEA that included:

- an assessment of the project' s geology and exploration leading to a release of a Mineral Resource estimate;
- an assessment of the geotechnical data and recommendation for geotechnical design guidelines;
- an preparation of open pit and underground mine designs and schedules;
- an assessment of the metallurgical considerations and preparation of plant design and metallurgical performance assumptions;
- an assessment of the hydrogeology and hydrological aspects;
- an assessment of the infrastructure requirements;
- an assessment of the environmental and permitting considerations;
- an estimate of operating and capital cost inputs; and
- techno-economic modelling.

SRK' s scope included consideration for processing mineralization at Newmarket' s' Union Reefs Processing Plant (including oxide mineralization) and through a stand-alone plant constructed on-site at Maud Creek (excluding oxide mineralization) to produce a saleable gold concentrate and gold Dore.

A previous Technical Report, Bremner, P and Edwards, M, 2012. Report on the Mineral Resource and Mineral Reserve of the Maud Creek Gold Project, excluded Oxide mineralization, assumed processing including a Bacterial Oxidation plant (BIOX®) and sale of gold Dore.

This Technical Report summarises the findings of the assessment of the option of processing mineralization at the Union Reefs Processing Plant. The techno-economic results of the PEA are presented in Table ES-1

Table ES-1: Summary of PEA Results

Parameter/ Result	Units	Quantity
Gold Price	AUD/oz	1,550
Exchange Rate	AUD: USD	0.77
Gold Price	USD/oz	1,200
Mine Life	Years	9.5
Mineral Inventory	'000 t	3,911
Diluted Gold Grade	g/t	4.2
Contained Gold	koz	528
Gold Recovery (oxide/transitional)	%	85
Gold Recovery (sulphide)	%	95
LOM Recovered Gold	koz	496
Production Rate	ktpa	500
Average Annual Gold Production	koz	52
Peak Annual Gold Production	koz	70
Annual Tonnes Concentrate (Dry)	kt	30
Concentrate Grade	g/t con	45
LOM Operating Cost	AUDM	408
LOM Cash Operating Cost	AUD/oz	822
Total Operating Costs/tonne Milled	AUD/t	105
Net Revenue (less selling expenses)	AUDM	725
Pre-Production Capital cost	AUDM	42
Sustaining Capital Cost (LOM)	AUDM	14

It is important to note that the PEA is preliminary in nature that it includes Inferred mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and there is certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have documented economic viability.

Based on the results of the comparisons of the economics, Table ES-2, and infrastructure considerations associated with both options the decision was taken to present the option of utilising the Union Reefs Processing Plant as the preferred option.

The economic analysis treats the Northern Territory Royalty as a Post-Tax cost, as is it is influenced by the NT Royalty Net Negative Value. SRK notes that Newmarket Gold NT Holdings Pty Ltd carries tax losses that have not been included in the economic analysis. Based on the integration of the Project this has the potential to further impact the value of the Project. The Northern Territory Royalty with no "losses carried forward" is $107 per ounce. The estimated available tax losses and NT Royalty Net Negative Value as at 31 December 2015, provided by Newmarket Gold is:

Income tax non-capital losses AUD229.8M

NT Royalty Net Negative Value AUD151.0M.

Table ES-2: Comparison of Options

	Units	Union Reefs Plant Quantity	Stand-alone Plant Quantity
Mine Life	years	9.5	8.0
Mineral Inventory	'000 t	3,911	3,460
Gold Grade	g/t	4.2	4.1
Contained Gold	koz	528	458
LOM Recovered Gold	koz	496	433
Gold Price	AUD/oz	1,550	1,550
Exchange Rate	AUD: USD	0.77	0.77
Gold Price	USD/oz	1,200	1,200
LOM operating cost	AUDM	408	378
Net Revenue (less selling expenses)	AUDM	725	633
Capital cost	AUD M	56	121
Pre-tax Cashflow	AUDM	261	134
Pre-tax NPV (5)	AUDM	201	89
After Tax Cash Flow	AUDM	182	91
After Tax NPV (5)	AUDM	137	55

Property Description and Location

The Maud Creek Gold project (Maud Creek or the Project) is located within the Pine Creek region of the Northern Territory of Australia, 20 kilometers east of Katherine. Previous mining activities at Maud Creek have been limited to open pit mining during 2000 when the owner was AngloGold.

The project comprises a total of 4 mineral titles (all granted), and the deposit is located wholly within tenement ML30260 which is held 100% by Newmarket Gold. Maud Creek is located at latitude 14°26' 41" south and longitude 132°27' 10" east.

Accessibility, Climate, Local Resources, Infrastructure and Physiography

Access is gained to the Project from Darwin by travelling south for 314 road kilometers along the sealed Stuart Highway to the town of Katherine.

Darwin has a population in excess of 129,000 and is the capital city of the Northern Territory. It is the administrative centre of the Northern Territory government and a major transportation hub, with an international airport and deep water port and the Adelaide to Darwin transcontinental railway terminating at the East Arm port.

Katherine is a regional centre with a population of approximately 9,800 and enjoys excellent infrastructure, services and communications. This is the closest centre of population to the Maud Creek project. Nearby regional mining communities of Pine Creek (with a population of 450) and Adelaide River (population of 200) support the Burnside, Maud Creek and Moline gold projects.

The major land use is grazing on native pastures and traditional Indigenous uses with some horticulture, grazing on modified pastures and nature conservation. The region has undergone some clearing (approximately 167,000 ha) for these developments The vegetation of the Maud Creek area consists largely of woodlands and open woodlands (predominant species - Eucalypts) that have been degraded by the impacts of cattle, buffalo and wild donkeys. No rare, threatened or endangered species have been identified in the area.

In the Maud Creek area, the terrain is flat lying to undulating. Ephemeral streams transect the project area and drain into the westward flowing Katherine River that flows all year.

The Top End of the Northern Territory has a tropical monsoon climate characterized by two distinct seasonal patterns: the ‘wet’ monsoon and the ‘dry’ seasons. The wet season generally occurs from November through to April and the dry season between May and October. Almost all rainfall occurs during the wet season, mostly between December and March, and the total rainfall decreases with distance from the coast.

History

Gold was initially discovered in the Maud Creek area in 1890 and a small plant was set up but ultimately abandoned in 1891. This is now called the Chlorite Hills and O’ Shea’ s area.

The area was re looked at from 1932 34 when 400 tonnes of ore produced 540 ounces of gold. Mining was from about 20 shallow shafts and small holes that were 6 12 meters deep with horizontal workings from 15 30 meters in length in the Chlorite Hills and O’ Shea’ s area.

Interest in the Maud Creek area was rekindled in the 1960s during an assessment of the mineral potential of the Top End of the Northern Territory. This study was prompted by the discovery of significant uranium mineralization in the nearby South Alligator River valley in the mid 1950s.

The Maud Creek project was owned by a number of companies until the acquisition of the Project by GBS Gold in December 2006. Substantial drilling, in the order of 66,000 90,000 m of RC and diamond drilling, is reported on the Project area during the period 1966 – 2006, oriented toward gold exploration.

AngloGold acquired rights to mine the oxide zone of the Main Zone deposit at Maud Creek and be processed at the Union Reefs Processing Plant. Mining operations were conducted during 2000. A total of 173,581 tonnes at 3.32 g/t Au produced 18,527 ounces. Ore was trucked from Maud Creek to the Union Reefs Processing Plant.

An agreement to acquire a number of properties, including the Maud Creek property, was entered into on June 19 2009 from GBS Gold International Inc. (GBS Gold) (In liquidation). GBS Gold operated the Tom’ s Gully and Brock’ s Creek underground gold mines, mined several open pit gold deposits and operated two gold Processing Plants, one at Tom’ s Gully, the other at Union Reefs, near Pine Creek, Northern Territory, until September 2008, when administrators were appointed.

On November 6 2009, the mining tenements including the Maud Creek Property were registered in the name of Crocodile Gold Australia Pty Ltd, a subsidiary of Crocodile Gold, which became Newmarket Gold in July 2015.

Geological Setting and Mineralization

The Maud Creek Gold deposit is located in the south-eastern part of the Pine Creek Geosyncline, within the Gold Creek Fault Zone, which forms the contact between mafic tuffs of the Dorothy Volcanics to the east and sedimentary rocks of the Tollis Formation to the west.

The Tollis Formation is the youngest member of the Finniss River Group, has limited aerial extent and consists of a succession of interbedded mudstone, slate, metagreywacke and minor felsic volcaniclastic shales. The Dorothy Volcanic Member consists of volcanic tuff with minor interbedded zones of sediments.

The north-south trending Gold Creek Fault Zone and primary Maud Creek mineralized zones dip steeply to the east. The deposit is roughly bound to the east by the Maud Creek Dolerite, which also exhibits mineralization at the tuff/dolerite contact and to the north by a small andesite body located at the contact between the sandstone and tuff (Maud Creek Contact Fault). To the south of the deposit a major east-west structure with sinistral strike-slip movement has been interpreted. Eight faults have been identified in the Maud Creek deposit area and generally exhibit reverse movement, with limited offsets in the range of meters.

The Maud Creek Contact Fault is filled with quartz stockwork veins; three vein lodes have been modelled for the Maud Creek deposit; the primary contact vein, upper contact vein and lower contact vein. The primary vein is strongly associated with the sandstone/tuff contact but does not strictly follow the boundary. Therefore, it has been modelled as an 'overprinting' volume onto the sediment, tuff, dolerite and andesite lithology wireframes. Mineralization in the east at the tuff/dolerite contact generally form steeply dipping discrete lenses with limited continuity.

Outside of and adjacent to the Maud Creek Contact Fault mineralization are many intercepts carrying similar grades to those within the vein itself; these extend up to 25 meters into the hanging wall and to a lesser extent into the footwall. In addition, a greater than 0.1 g/t Au halo can be observed up to 50 meters into the hanging wall and occasionally in the footwall.

Deposit Types

A variety of genetic models have been postulated for the formation of gold deposits in the Pine Creek Geosyncline. Gold and base metal mineralization is commonly associated with granite intrusions and are often been classified as high temperature contact aureole deposits. A secondary host rock control has also been suggested due to the association of gold mineralization with carbonaceous metasedimentary rocks. More recently, authors have argued that gold mineralization is structurally controlled; occurring in brittle ductile structures at the greenschist amphibole facies boundary and hence has an epigenetic origin.

Accepting that gold deposits of the Northern Territory have a structurally controlled mesothermal setting, then on the basis of host rock and mineral association they can be divided into seven types:

- Gold quartz veins, lodes, sheeted veins, stockworks, saddle reefs (Pine Creek Orogen);
- Gold ironstone bodies (Tennant Inlier);
- Gold in iron rich sediments (Pine Creek Orogen, Tanami);
- Polymetallic deposits (Iron Blow, Mt Bonnie);
- Gold PGE deposits (South Alligator River area);
- Uranium gold deposits (Pine Creek Orogen, Murphy Inlier); and
- Placer deposits.

Of these types, Maud Creek aligns with the gold-quartz veins, lodes, sheeted veins, stockwork deposit type. Five main types of mineralization have previously been recognized within the Pine Creek Orogen. These include:

- Sheeted and stockwork quartz vein systems located along major anticlinal hinges;
- Sediment hosted stratiform gold mineralization and quartz sulphide vein hosted stratabound gold mineralization in cherty ironstone and carbonaceous mudstone;
- Stratiform, massive to banded, sulphide silicate carbonate mineralization ;
- Sediment hosted stratiform and stratabound gold mineralization in cherty, dolomitic and sulphidic shales; and
- Sheeted or stockwork quartz feldspar sulphide veins.

Of these mineralization types, Maud Creek is consistent with stockwork quartz-feldspar-sulphide veining hosted at the contact of either sandstone/tuff or tuff/dolerite units.

Mineral Resource Estimates

The Mineral Resources are stated here for the Maud Creek deposit with an effective date of 15 March 2016 and were previously reported by Fairfield and Kentwell (2016).

The Maud Creek deposit consists of open pit and underground resources presented in Table ES-3 and ES-4. All relevant diamond drillhole samples, available as of April 2015 for the Maud Creek deposit were used to inform the estimate. The estimation methodology utilised was Ordinary Kriging (OK) to estimate gold and arsenic using hard domain boundaries.

Table ES-3: Open pit Mineral Resource above 950 mRL at 0.5 g/t Au cut-off – base case

Mineral Resource Category	Inventory (kt)	Gold Grade (g/t)	Contained Metal (koz Au)
Measured	1,070	5.6	190
Indicated	1,100	2.1	75
Measured and Indicated	**2,170**	**3.8**	**268**
Inferred	530	1.4	25

It should be pointed out the Mineral Resource estimate is categorized as Measured, Indicated and Inferred as defined by the CIM guidelines for resource reporting. Mineral resources do not demonstrate economic viability, and there is no certainty that these Mineral Resources will be converted into mineable reserves once economic considerations are applied. The Measured, Indicated and Inferred Mineral Resource estimate has been prepared in compliance with the standards of NI 43 – 101 by Danny Kentwell, FAusIMM.

Notes to Table ES-3:

1. CIM definitions followed for classification of Measured, Indicated, and Inferred Mineral Resources.
2. Mineral Resources estimated as of 15 March 2016.
3. Mineral Resources stated according to CIM guidelines.
4. Totals may appear different from the sum of their components due to rounding.
5. Reported at a 0.5 g/t cut-off grade.
6. The open pit Mineral Resource is exclusive of the underground Mineral Resource.
7. The Mineral Resource estimation was performed by Danny Kentwell FAusIMM fulltime employee of SRK Consulting, who is a Qualified Person under NI 43-101.

Table ES-4: Underground Mineral Resource below 950 mRL at 1.5 g/t Au cut-off – base case

Mineral Resource Category	Inventory (kt)	Gold Grade (g/t)	Contained Metal (koz Au)
Measured	-	-	-
Indicated	4,330	3.2	456
Measured and Indicated	**4,330**	**3.2**	**456**
Inferred	1,450	2.7	124

It should be pointed out the Mineral Resource estimate is categorized as Indicated and Inferred as defined by the CIM guidelines for resource reporting. Mineral resources do not demonstrate economic viability, and there is no certainty that these Mineral Resources will be converted into mineable reserves once economic considerations are applied. The Measured, Indicated and Inferred Mineral Resource estimate has been prepared in compliance with the standards of NI 43-101 by Danny Kentwell, FAusIMM.

Notes to Table ES-4:

1. CIM definitions followed for classification of Measured, Indicated, and Inferred Mineral Resources.
2. Mineral Resources estimated as of 15 March 2016.
3. Mineral Resources stated according to CIM guidelines.
4. Totals may appear different from the sum of their components due to rounding.
5. Reported at a 1.5 g/t cut-off grade.
6. The underground Mineral Resource is exclusive of the open pit Mineral Resource.
7. The Mineral Resource estimation was performed by Danny Kentwell FAusIMM fulltime employee of SRK Consulting, who is a Qualified Person under NI 43-101.

In SRK' s opinion, based on the depth and distribution of the mineralization open pit and underground mining could be viable options for extraction.

In assessing the criteria for reasonable prospects of economic extraction both open pit and underground scenarios were considered. With respect the scattered lower grade mineralization contained within the near surface a simple pit optimisation using the optimistic parameters at twice the current gold spot price did not generate a pit of practical size on the eastern domains. All material in the eastern domains is not considered to have reasonable prospects of economic extraction and does not appear in the Mineral Resource.

With respect to the underground potential the grade is reasonably consistent down to approximately 650 mRL below which it drops significantly. All material below 650 mRL is not considered to have reasonable prospects of economic extraction and does not appear in the Resource.

Mining

Based on the geological review of the deposit, SRK has prepared a mining design and schedule based on a combination of conventional open pit and underground mining operations. The proposed mill feed estimated from SRK' s design is presented in Table ES-5 and a breakdown by classification in Table ES-6.

Table ES-5: Proposed Mill Feed

	Inventory (kt)	Grade (g/t)	Contained Metal (koz Au)
Open Pit	634	5.1	104
Underground	3,276	4.0	423
Total	**3,911**	**4.2**	**528**

Table ES-6: Underground Mining Inventory by Resource Classification

Resource Classification	% of Feed	Diluted Mined Tonnes and Grades		
		Inventory (kt)	Grade (g/t Au)	Contained Metal (koz Au)
Measured	20	679	6.1	132
Indicated	70	2,306	3.9	290
Inferred	10	291	2.8	26
Underground Mining Inventory	**100**	**3,276**	**4.0**	**423**

It is important to note that the PEA is preliminary in nature that it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and there is certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have documented economic viability.

Geotechnical

In conjunction with the Mineral Resource review, a comprehensive review of the available data and analysis to determine open pit and underground geotechnical design guidelines.

An assessment of overall slope angles and underground mining parameters has been undertaken using geological and geotechnical drilling data supplied by Newmarket Gold. The analysis provides good early-stage design guidelines of the geotechnical properties of the rock mass. The typical geotechnical conditions on site can be summarised as follows:

The Hanging wall Tuffs are typically massive but may be locally bedded. Hanging wall tuffs are also affected by the numerous shears present in the Hanging wall, resulting in reduced strength, increased fracture frequency and graphitic and/or chloritic alteration of the rock mass.

The Footwall Sediments consist of low to medium strength thinly bedded or laminated mudstone and siltstone, and medium to thickly bedded sandstone . Zones of intense shearing with chlorite and graphite alteration occurring in the 5 to 10 meters below the mineralized zone where the sediments are commonly black, highly graphitic and/or chloritic, very weak and fissile.

The competency of the mineralized zone can be expected to be variable with competent, partially silicified mineralized zones separated by zones of intensely sheared rock.

The distribution of the various fault configurations is not understood at this stage and this should be one of the main focus for subsequent field investigations.

The absence of suitable data has led to low-confidence in the geotechnical conditions. Additional data is required to improve confidence and refine decisions on mining methods and the mine design. The mining method studies are linked to the decision on the location of the Processing Plant and the availability of pastefill.

The proposed base case is utilisation of the Processing Plant being located at Union Reefs; as such a waste fill mining option has been considered with a "bottom-up" mining sequence.

Hydrogeology

SRK reviewed the findings and basis of previous reports and considers that them to be fit for purpose. Based on the absence of the underlying data, the purpose and stage of this study it was not appropriate to undertake additional hydrogeological modelling.

Should the study progress, additional data is required and should be incorporated with the revised structural / geological model to develop an updated numerical model. This would either build on the existing model, if data can be made available or a new model would be constructed. Specifically the updated model would confirm / address dewatering rate assumptions and would seek to collect data at depth. There is a distinct lack of knowledge at depth in the existing model.

Water management of the site has been incorporated as a key factor, to keep clean water clean and direct the contact water to appropriate containment systems. A preliminary analysis has been completed to prevent flows associated with the 1-in-1000 year ARI event from encroaching the open pit.

The designed infrastructure includes a flood protection bund to the east of the pit, a diversion channel to maintain Gold Creek to the east of the pit has been conceptually designed and a dedicated water pond for the excess water.

Based on the available data, the preliminary water balance indicates the water demand is met throughout operations. There is excess water in the system to be treated and managed on-site. SRK has modelled the use of a water treatment plant as opposed to an evaporation facility.

Additional study work will be required once a greater understanding of the water quality and quantity is determined.

Mineral Processing & Metallurgical Testwork

An extensive program of metallurgical testing was carried out from 1994 through to 2006 at reputable and suitably experienced laboratories. Testing was undertaken at both batch and pilot scale and including variability testing. Part of the focus of testing was on downstream oxidation processes on the refractory and preg-robbing Maud Creek mineralization, such as bio-oxidation (BIOX®) and the GEOCOAT® process. Direct cyanidation leaching of mineralization and concentrates was tested on the fresh (sulphide) mineralization with poor results and was eliminated as a potential processing route. A number of engineering studies were undertaken in conjunction with this testwork.

Metallurgical testing has shown the fresh mineralization, the bulk of the Project tonnage, to be moderately hard and abrasive, to have variable levels of gravity gold recovery, to be refractory and preg-robbing in nature but responsive to simple flotation techniques ‐ demonstrating high gold recoveries in excess of 95%. Total recovery is consistently high irrespective of gravity recovery. The flotation concentrate has sufficient grade to be classified as a gold concentrate for the purposes of importation into China (> 40 g/t). It is noted that part of the gold is associated with arsenopyrite and as a result, arsenic grades in the concentrate are elevated at approximately 3.6% . There is no arsenic restriction on the importation of gold concentrates.

The Maud Creek mineralization has been subject to extensive metallurgical testing, adequate to support this PEA. Some additional testwork is recommended by SRK for the next level of design to further increase confidence of the study if new drill core becomes available. This includes variability testing of metallurgical behaviour at depth (gravity and comminution particularly), flash flotation sighter tests and potentially some further assessment of the oxide and transitional mineralization. The customers will likely require a sample of the final flotation concentrate for their own testing. This will necessitate additional metallurgical testwork at some point and can be done in conjunction with the other recommended testing requirements. Some assessment of reagent contamination of the flotation circuit (specifically cyanide) and the impact on flotation recovery is also important now that the Project base case assumes processing is at Union Reefs.

Maud Creek oxide mineralization can be processed through the existing Union Reefs CIP plant and can be suitable treated. The transitional feed, which makes up just a small portion of the overall LoM tonnage but a significant part of the first years feed will have more variable metallurgical behaviour. Transitional mineralization can be processed through the existing oxide circuit and/or the new flotation circuit to optimize recovery. Sulphidation flotation testing is a possibility to optimize transitional mineralization recovery.

The current PEA has reviewed the previous metallurgical testwork data and supporting engineering studies. From this body of work, a simple, conventional, inexpensive and low-risk processing circuit has been selected, allowing flexibility with respect to circuit configuration and downstream processing. It originally assumed a standalone Maud Creek Processing Plant processing fresh mineralization only. The engineering undertaken as part of this PEA reflects this. Subsequently the Project assumption changed to processing mineralization through a modified Union Reefs Processing Plant as the base case. Any preliminary engineering gaps are largely associated with this change.

The base case presented for this study is a Brownfield upgrade to the existing and operating Union Reefs Processing Plant. This would see the addition of flotation and concentrate dewatering circuits, producing a gold sulphide concentrate for sale. Part of the concentrate production could be processed at Newmarket Gold’ s Fosterville BIOX® plant if spare capacity is available. A gravity gold concentrate will also be produced. The preference would be to smelt it into gold doré on site but alternatively it could be smelted and refined in Australia at one of the established gold refiners.

A detailed integration study has not been undertaken into the Union Reefs processing option. This will be undertaken at the next Phase of the Study. However it is expected to be relatively simple, quick and straight forward. It presents a significant opportunity to processing of Maud Creek mineralization.

At the Union Reefs Processing Plant, the ore is crushed and then milled in closed circuit with hydrocyclones. A separate gravity circuit recovers gravity recoverable gold from the ball mill discharge. The fine milled product from the cyclone overflow is pumped to a new flotation circuit where a high grade low tonnage gold bearing concentrate is produced from the sulphide minerals, leaving a gangue non-sulphide residue that can be disposed of into tailings. The concentrate is dewatered through thickening and filtration before being bagged, stored in shipping containers and transported by road, then ship to customers in China.

Downstream processing options for refractory gold mineralization and concentrates were considered at a preliminary level. The direct smelting of flotation concentrates option has been selected as the preferred option.

There are a number of advantages associated with leveraging the existing Union Reefs Processing Plant rather than building a standalone Processing Plant at Maud Creek including;

- Lower processing and infrastructure capital costs;
- Lower first fill costs;
- Low additional sustaining capital costs;
- Simpler and quicker project implementation, reduced technical risk;
- Lower water demand at Maud Creek;
- Less onerous approvals for existing processing facility;
- Lower overall operating cost;
- Production creep opportunities with existing facility;
- Simple and cheap future expansion to meet any increase in mining production;
- Ability to process oxide mineralization, so increasing the LoM of Maud Creek;
- More flexibility in processing, including transitional mineralization being able to be processed through the cyanide leach or flotation circuits or possible both, (higher gold recovery);
- Extensive tailings storage capacity already available (lower capital cost); and
- Extend the LoM of the Union Reefs/Cosmo underground mine operation.

The key additional cost is haulage of mineralization. Other key considerations include the required approvals required for this haulage and obtaining the social license to operate.

Project Infrastructure

The Maud Creek Gold Project surface facilities are representative of a modern and conventional underground mining operation. The site comprises the following:

- Office and administration complex, including change house;
- Store and laydown facilities;
- Heavy underground equipment workshop;
- Temporary surface stockpiles and waste stockpile area;

Maud Creek Open Pit Mine and portal;

Ventilation exhaust raises;

Ventilation intake raise;

Raw water storage to manage rainfall runoff; and

ROM haul road along the existing access road to the town of Katherine.

The Union Reefs Processing Plant was commissioned in 1994 with an upgrade in 1998. Key infrastructure at the plant includes:

Office and administration complex;

Store and laydown facilities;

Gravity and CIL Processing Plant and associated facilities;

Process plant workshop;

Electrical substations and transformer supplied by 66 kV power line;

Reagent storage;

Laboratory;

Vehicle wash- down area;

Tailings storage facilities;

Run of Mine stockpiles; and

Core processing facility.

The processing team will continue to be provided from the Cosmo Underground Mine accommodation village located 53 km north of Union Reefs, or through residential employment based in the nearby towns of Pine Creek, Adelaide River and Katherine. The Cosmo Accommodation Village is managed by an independent contractor. Maud Creek mining accommodation will be provided for in Katherine or the local surrounds.

Ore Haulage

ROM mineralization will be hauled 144 km from the Maud Creek Mine to the existing Union Reefs Processing Plant. The access road form Katherine to Maud Creek will be upgraded to allow for heavy haulage. A preliminary study into light and heavy vehicle movements on the proposed haul route in and around Katherine has been generated using publicly available 2014 vehicle movements on the Stuart Hwy.

It is anticipated that the mine haulage operations will utilise quad semi-trailers, which will involve travel from Maud Creek to Union Reefs (loaded) and back again (empty) on a daily basis; therefore, the mining operations will require approximately 26 road train movements per day (13 full and 13 empty). It is proposed that haulage from the site and traffic to the site will occur only during daylight hours. This would nominally be from 6:00AM to 6:00PM. This will occur throughout the year during both the wet and dry seasons. Confirmation of allowable number of trailers and load through town centres is required at the next phase of the pre-feasibility study. Any reduction will require a corresponding increase in truck movements each day.

The increase in traffic on the Stuart Highway resulting from the mining operations at Maud Creek would be 2.4% north of Katherine, 6.9 percent south of Katherine and 0.7% within the Katherine Township. More significantly, in terms of heavy vehicle movements, the 26 quad road train movements per day represents a 24.6% increase north of Katherine and a 26.9% south of Katherine.

The major safety impacts of these road trains will be at the two intersections where the road trains turn onto or off the Stuart Highway and ensuring adequate overtaking opportunities for cars.

In July 2015, the Northern Territory Government started the planning on a long-term heavy vehicle alternate route that bypasses the central business area of the township. The alternate heavy vehicle route study was proposed following a number of road safety issues in the main street, as it currently accommodates a mix of pedestrians, local and tourist traffic, and heavy vehicles. The alternate heavy vehicle route would dramatically reduce the impact of the Maud Creek haulage operations. Government and public consultation and approval will be a key element to the viability of processing of Maud Creek mineralization at the Union Reefs Processing Plant.

Power

A preliminary power supply options assessment has been undertaken to supply the approximately 2.0 MW of power demand for the underground mining operations. It considered mains power and site generated power (diesel and/or natural gas). Due to the limited demand from mining and ramp up in requirement, it is proposed that electrical power is provided via on site diesel generators supplied, owned and operated by an independent power provider (IPP).

Power at the Union Reefs Processing Plant is supplied via the 66 kV Darwin-Katherine distribution network. There is sufficient capacity to meet the modifications to the plant. The difference in power demand operating the existing CIL circuit or the new flotation circuit would not be significant.

Concentrate Transport & Contracts

The PEA identifies the potential for 36,500 tonnes (dry) of gold concentrate to be produced annually from the Maud Creek deposit. The concentrate will be stored in 1.5 tonne bulk bags which are then loaded into shipping containers ready for transportation. Containers are trucked from the Union Reefs Processing Plant to the Darwin Port, approximately 220 km. From there they are shipped to Chinese customers. At Darwin Port, the containers are stacked until required for ship loading with fortnightly sailings scheduled. Preliminary costs assume Dalian Port as the destination port in China. This has a transit time of 29-36 days via Shanghai or Kaoshiung.

Indicative terms have been provided to Simulus Engineers for the sale of a gold concentrate to Shangdong Zhong Guo (China Gold Shandong) for processing at their Yantai Gold Smelter. Payment terms are USD10/t for processing and payment for 95% of the contained gold. Similar terms were provided by Baxville (Beijing) Minerals Trading Ltd. Additional smelting terms were provided verbally by Australian gold concentrate producers to allow further benchmarking.

Other inquiries have been issued to potential customers and traders. Indicative terms for the sale to another direct exporter of concentrate (i.e. to consolidate with their concentrate) who sell to Guoda Gold Co Ltd, or sale to an Australasian Pressure Oxidation (POX) facility with additional capacity have been referenced from other similar gold concentrates projects and operations in Australia. Informal discussions have also been held with the owners of a number of Australian third party refractory gold operations for sale of a generic gold concentrate and to assess available capacity at those facilities but discussions have not been progressed to any great extent. It has been used mainly in the process of elimination of the bulk of these options. Further investigations into the remaining options can be undertaken at the next stage of study if deemed appropriate.

The sale of the gold concentrate to China remains the current base case. No formal concentrate discussions or sales contracts have been entered into at this stage of study. Gravity gold will be recovered into gold doré on site and sold to established Australian refiners.

Economic Analysis

The key economic assumptions are provided in Table ES-7. A summary of the project economics are presented in Table ES-8. SRK notes the gold price at the time of reporting is over AUD1,600/oz.

SRK has applied a discount rate of 5% on the basis that Newmarket Gold is a Canadian based company and will source funding from US/ Canadian markets. An increase in the discount rate to 8% reduces the After Tax NPV by AUD20M.

It is important to note that the PEA is preliminary in nature that it includes Inferred mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and there is certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have documented economic viability.

Newmarket Gold NT Holdings Pty Ltd carries tax losses that have not been included in the economic analysis. The estimated available tax losses as at 31 December 2015, provided by Newmarket Gold is:

Income tax non-capital losses AUD229.8M

NT Royalty Net Negative Value AUD151.0M.

Table ES-7: Economic Assumption Criteria

Description	Units	Quantity
Gold Price	Gold AUD/oz	1,550
Exchange Rate	AUD: USD	0.77
Gold Price	USD/oz	1,200
Discount Rate	%	5

Table ES-8: Project Economics

	Units	Union Reefs Plant Quantity	Stand-alone Plant Quantity
Mining			
Mine Life	years	9.5	8.0
Mineral Inventory	‘000 t	3,911	3,460
Gold Grade	g/t	4.2	4.1
Contained Gold	koz	528	458
Open Pit Tonnes Ore	kt	634	168
Open Pit Gold Grade	g/t	5.12	6.1
Open pit Contained ounces	koz	104	33
Total Waste Mined	‘000 t	5,000	1,358
Strip Ratio (O/P)	t:t	8	8
Underground Tonnes Ore	‘000 t	3,276	3,292
Underground Gold Grade	g/t	4.02	4.02
Underground Contained ounces	koz	423	425
Processing			
LOM Tonnes Milled	‘000 t	3,911	3,460
Production Rate	ktpa	500	500
Average Annual Gold production	koz	52	54

	Units	Union Reefs Plant Quantity	Stand-alone Plant Quantity
LOM Recovered Gold	koz	496	433
Metallurgical Recovery			
Transitional to Float	%	65	65
Transitional to Gravity	%	20	20
Fresh to Float	%	75	75
Fresh to Gravity	%	20	20
Annual Tonnes Concentrate (Dry)	kt	30	32
LOM Tonnes concentrate (Dry)	kt	285	253
Concentrate Grade	g/t con	45	45
Economics			
Gold Price	AUD/oz	1,550	1,550
Exchange Rate	AUD: USD	0.77	0.77
Gold Price	USD/oz	1,200	1,200
LOM operating cost	AUDM	408	378
LOM operating cost	AUD/t milled	105	109
LOM operating cost (Payable)	AUD/oz	856	910
LOM operating cost (Payable)	USD/oz	662	703
LOM Payable (Saleable Gold)	koz	477	416
Net Revenue (less selling expenses)	AUDM	725	633
Capital cost	AUD M	56	121
Pre-tax Cashflow	AUDM	261	134
Pre-tax NPV (5)	AUDM	201	89
Pre-tax IRR	%	116	28
Payback Period (pre-tax)	Qtr	6	15
After Tax Cash Flow	AUDM	182	91
After Tax NPV (5)	AUDM	137	55
After Tax IRR	%	80	20
Payback Period (After Tax)	Qtr	6	18

Sensitivity

The key drivers are the revenue assumptions (Grade, Recovery and Gold Price) and the operating cost assumptions.

SRK notes that Capital Cost estimates used for the sensitivity analysis exclude the capital development as the cost driver for this is the unit operating costs. Impact on the capital development costs is included in the operating cost sensitivity.

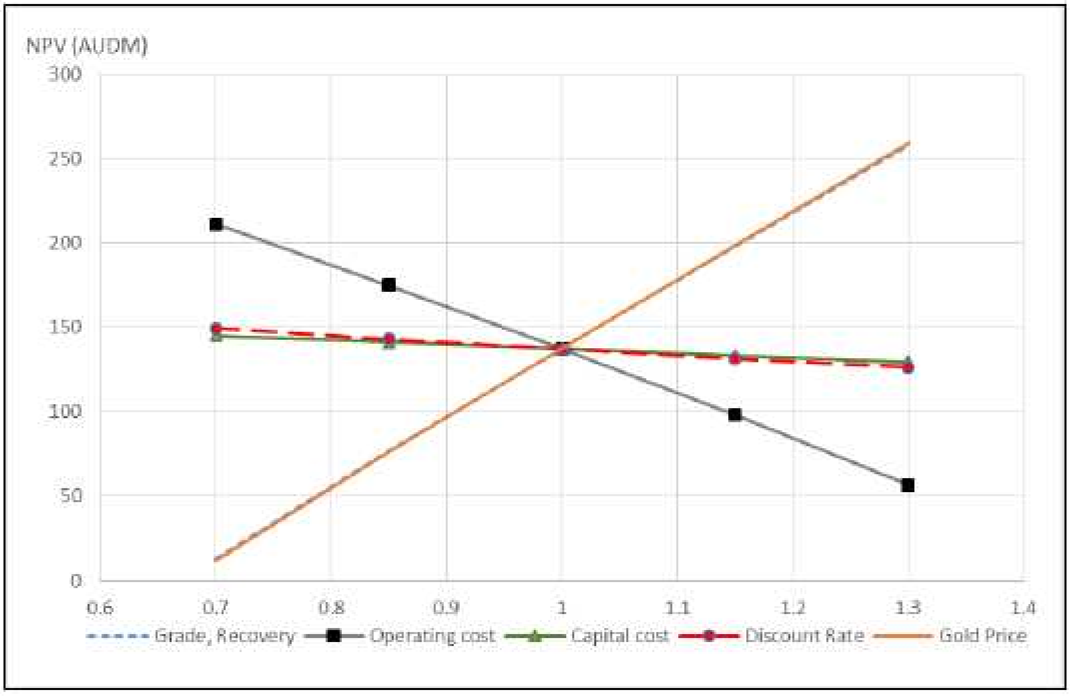


Figure ES-1: Sensitivity Analysis

Sensitivity to the Gold Price is presented in Table ES-9.

Table ES-9: Gold Price Sensitivity

Gold Price (AUD/oz)	1,400	1,450	1,500	1,550	1,600	1,650	1,700
Pre-Tax NPV5% (AUDM)	145	163	182	201	220	239	257
Pre-Tax IRR%	85	95	106	116	127	138	150
After-Tax NPV5% (AUDM)	98	111	124	137	150	163	177
After-Tax IRR%	59	66	73	80	87	94	102

Interpretation and Conclusions

It is important to note that the PEA is preliminary in nature that it includes Inferred mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and there is certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have documented economic viability.

Based on the economic findings of the PEA, SRK concludes that the Project has merit and that Newmarket Gold consider progressing the study to the next level of detail.

In doing so SRK provides recommendations that Newmarket Gold consider progressing the additional work programs to better understand the key technical risk areas to support future evaluations.

The 2015 geological model is considered by SRK to be more robust in terms of its geological basis and this has led to a slightly higher grades but a reduction in contained gold. Only further drilling can define true connectivity of the mineralization in widely spaced areas. Discussion of the risks and opportunities in the Mineral Resource model is presented in Table ES-10.

Table ES-10: Mineral Resource Model Risks and Opportunities

Project Element	Economic Risk Level	Comment	Opportunity
Database - Exploration data	Low	Historical and recent data have been re- collated and re-validated for this Mineral Resource estimate.	
Assaying	Low	QAQC for recent and older assaying shows no material issues. Arsenic assaying has incomplete coverage.	Additional assaying for Arsenic may be beneficial depending on the processing method.
Surveying	Low	Both collar surveys and downhole surveys completed to a high level of accuracy for recent drilling. Representative collars resurveyed for older drilling with no significant discrepancies.	
Geology	Low	Detailed logging and interpretation together with evidence from both regional structural features and detailed in pit mapping informs the geological understanding	Additional drilling may be able to add detail to the interaction of structures controlling mineralization at depth.
Geological modelling	Low	A detailed structural and lithological model has been built and incorporated into the estimation domain construction.	Additional drilling may be able to add detail to the interaction of structures controlling mineralization at depth.
Resource Estimation	Low	Ordinary kriging cross checked and validated with theoretical grade tonnage curves and alternative search parameters has been used.	The project may benefit from simulation studies or non-linear estimates if detailed studies at selective mining unit block sizes are required in the future.

The interpreted level of confidence of the available Geotechnical data streams were rated subjectively using a 5-point rating scale of Very Low - Low - Moderate - High - Very High has been used (refer Table ES-11).

Table ES-11: Qualitative risk assessment of study components

Data	Confidence level
Empirical: rock mass characterisation	Low
Structural: major structures	Low to Moderate
Structural; rock mass structures	Low
Rock mass strengths	Very low
Rock material strengths	Very low
Groundwater conditions	Very low
Slope angle recommendations	Low

At this stage of the project development, a Low confidence rating should be expected for all items, with all items requiring further investigation. Aspects for which no data is currently available or represent a key concern have been flagged as Very Low.

The geotechnical assumptions are key inputs to the mine design and schedule and must be better understood prior to advancing the study work.

Recommendations

Should the Project study work proceed, SRK recommends the following aspects are considered:

Mineral Resource

Infill drilling in the parts of the Mineral Resource currently classified as Indicated and Inferred would enable an upgrade of the Mineral Resource Classification. An approximate drilling meters to complete this from surface down to 850 mRL is provided in Table ES-9 assuming the drilling takes place from surface.

A number of sections of the geological model remain open down dip with good grades seen in the last hole down dip. Extension drilling is recommended to test these areas. The required drilling meters to complete these are shown in Table ES-12 assuming drilling from surface.

Table ES-12: Recommended Mineral Resource Drilling

Target	Current exploration status	Potential End of 2016 Status	Drilling Description	(meters)
Infill Drilling	Indicated and/or Inferred	Measured and/or Indicated	Increase confidence in estimated Mineral Resource	9,200
Extension drilling	Down dip or along strike from current Mineral Resource	Indicated and/or Inferred	Close off or extend Mineral Resource volumes	2,200

Geotechnical

The geotechnical recommendations, summarised in Table ES-13, includes the following:

Dedicated geotechnical drilling program to:

- Target areas that require additional data such as the footwalls sediments, and crown pillar area;
- Drilling proposed locations of LOM infrastructure – portals, declines, vent shafts etc.;
- Improve understanding of structural characteristics including continuity and orientation variability;
- To collect representative samples for laboratory testing;

Mapping and photogrammetry in current pit;

Geotechnical laboratory testing using NATA certified laboratories;

Detailed backfill design for potential mining methods (including CRF);

Potential stress measurement testing (AE or DRA); and

Numerical modelling of proposed mining layout and sequence (including crown pillar and central pillar sequence)

Table ES-13: Drill program targets and estimated depths

Description	Drillholes (No.)	Depth (m)	Total (m)
To investigate sediments at depth	4	600	2,400
To supplement data and sediments across mid depths of mine	4	500	2,000
Portal and boxcut investigation drill holes	3	50	150
To investigate Infrastructure located in sediments (can be separated into multiple shorter drill holes)	1	600	600
Open Pit geotechnical holes – will provide data on crown pillar area and upper sediments	6	150	900
Total meters for proposed program	18		6,050

Geometallurgy

Insufficient information was available from the metallurgical reports to create preliminary spatial domains for the physical processing parameters, such as grindability. The metallurgical report Core Process Engineering Report No. 140-001 outlined JK Drop Tests which had been conducted, resulting in a Bond ball Work Index of 18-19 kWh/tonne for the main lode. Previous geology reports, as well as a site visit to inspect the core, gave an indication of the mineralogy of the Maud Creek deposit, which has an impact on the hardness of the mineralization. From a geometallurgical perspective, between and within each mineralized domain there will likely be a range of hardness values which will need to be established. Quartz alteration at Maud Creek has been identified as varying between the primary lode (high percentage quartz veining), moderate (footwall and hanging wall lodes stock work veining) to low (sandstone and tuff country rocks). However variations in silica content within these lodes will definitely occur, as alteration boundaries are normally pervasive across lithology boundaries. A possible way to better define the hardness parameters is the use of proxies. If silica analysis is included in any future assay testing of the mineralized zones, this can be used to identify target areas for JK drop weight tests. A regression calculation between the A*b result, Bond work indices and other comminution parameters versus the silica content can then be determined, allowing a predictive model of hardness to be created.

Mineral analysis would need to be conducted to ensure the silica content is reflecting quartz alteration which has a high hardness (Mohs scale 7) and not feldspar (Mohs scale 5-6) or mica minerals (Mohs scale 2-3) which have lower hardness and varying crystallography, resulting in different grinding behaviours. The Geology of the Maud Creek Gold Deposit and Maud Creek Reconciliation report by AngloGold in 2000 identified plagioclase laths and alkali feldspar minerals in the Andesite unit located to the south of the deposit, but not within the mineralized lodes. However, sericite (a type of muscovite) and chlorite alteration (most likely the mineral clinochlore) are both quite prevalent in all three mineralized zones, so their prevalence would need to be established as well. Similarly, the extent of the hematite and marcasite alteration noted in all three mineralized zones (present as pseudomorphs of pyrite in the oxide zone) needs to be quantified, as all three will have moderate-high hardness (6-7), but also a brittle tenacity upon breakage, which will influence the ore grindability.

The Maud Creek mineralization is going to be very hard and abrasive so finding a correlation with proxies such as silica, for example, is recommended by SRK. Equally, determining correlations between arsenic, sulphur and gold to help generate the flotation relationship correlations, and approximated the amount of arsenic in the final concentrate is also recommended. The arsenic and sulphur contents will also be a good indicator of recovery.

Mining

Should the project proceed SRK recommends that a range of sensitivities/ scenarios are considered at a high level to inform the project team of the implications of these constraints. The sensitivities should include combinations of considering the open pit oxide as an ore source with both the underground preferences retained and removed. Conventional open pit mining techniques are proposed.

The opportunity exists to modify the design to interact with the open pit design but this will potentially have constraints on production continuity.

The mine design and schedule should be revised based on the findings of the geotechnical review.

Processing & Infrastructure

The following additional recommendations are provided based on the findings from the first phase of study and the new base case being to process mineralization at the Union Reefs Processing Plant, after modifications, rather than at a standalone processing facility:

Undertake a further review of the metallurgical testwork to understand the behaviour of the oxide mineralization;

Undertake testwork to understand the intensive leach behaviour of the gravity gold concentrate, generate flotation concentrate for customer testing and assessment, other minor testwork as discussed in the main body of this report;

Undertake a specialist comminution circuit capacity study processing Maud Creek mineralization through Union Reefs Processing Plant;

Undertake a more detailed integration study of a new flotation, dewatering and concentrate storage circuits into the Union Reefs Processing Plant in consultation with the Union Reefs and Newmarket Gold technical groups. This would include scope of work, opportunity upgrades at Union Reef, tie-in requirements and the campaign operating philosophy;

Update the capital cost to suit the next level of study with Union Reefs as the base case;

Update the operating costs to suit the next level of study with Union Reefs as the base case, using their power, labour, accommodation and other costs. Agreement on the share of benefits to Maud Creek and Cosmo projects is also required;

Develop cost of upgrading the access road from the Maud Creek Mine Site into Katherine;

Progress discussions with the local Government authorities to confirm in principal that haulage through Katherine and up to Union Reefs is acceptable – this is a potential fatal flaw to the base case processing option;

Confirm capital upgrade requirements attributable to Newmarket Gold (if any) to the main Stuart Hwy; and

Confirm road train maximum tonnage acceptable to Local and Territory Government.

Progress flotation concentrate off-take discussions, to provide more confidence in terms and conditions.

Table of Contents

List of Tables

List of Figures

2 Introduction

SRK Consulting (Australasia) Pty Ltd (SRK) was engaged by Newmarket Gold to undertake a study on the Maud Creek Gold Project (Maud Creek or the Project) and prepare a Technical Report summarising the findings of a Preliminary Economic Assessment (PEA).

In early July 2015, Newmarket Gold Inc. merged with Crocodile Gold Corp. (Crocodile Gold) to form a new Canadian, Toronto Stock Exchange listed gold mining company named Newmarket Gold that has 100% ownership of the Maud Creek Project.

The work and this Technical Report were prepared by SRK following the guidelines of the Canadian Securities Administrators' NI 43-101 and Form 43-101 F1.

The Mineral Resource statement reported herein was prepared in conformity with generally accepted Canadian Institute of Mining, Metallurgy, and Petroleum' s (CIM) Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines.

2.1 Scope of Work

The scope of this Technical Report documents the findings of the PEA that included:

- an assessment of the project' s geology and exploration leading to a release of a Mineral Resource estimate;
- an assessment of the geotechnical data and recommendation for geotechnical design guidelines;
- an preparation of open pit and underground mine designs and schedules;
- an assessment of the metallurgical considerations and preparation of plant design and metallurgical performance assumptions;
- an assessment of the hydrogeology and hydrological aspects;
- an assessment of the infrastructure requirements;
- an assessment of the environmental and permitting considerations;
- an estimate of operating and capital cost inputs; and
- technical economic modelling.

SRK' s scope included consideration for processing mineralization at Newmarket' s' Union Reefs Processing Plant (including oxide mineralization) and through a stand-alone plant constructed on-site at Maud Creek (excluding oxide mineralization) to produce a saleable gold concentrate and gold Dore.

A previous Technical Report, Bremner, P and Edwards, M, 2012. Report on the Mineral Resource and Mineral Reserve of the Maud Creek Gold Project, excluded Oxide mineralization, assumed processing including a Bacterial Oxidation plant (BIOX®) and sale of gold Dore.

Based on the results of the comparisons of the economics, Table ES-1, and infrastructure considerations associated with both options the decision was taken to present the option of utilising the Union Reefs Processing Plant as the preferred option.

This Technical Report summarises the findings of the assessment of the option of processing mineralization at the Union Reefs processing Plant.

It is important to note that the PEA is preliminary in nature that it includes Inferred mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and there is certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have documented economic viability.

2.2 Work Program

The Technical Report was assembled in Melbourne during the months of April 2015 to May 2016.

The Mineral Resource Statement reported herein was prepared in conformity with generally accepted CIM Exploration Best Practices and Estimation of Mineral Resource and Mineral Reserves Best Practices guidelines. This Technical Report was prepared following the guidelines of the Canadian Securities Administrators National Instrument 43-101 and Form 43-101 F1.

2.3 Basis of Technical Report

The purpose of this Technical Report is to present the geological review of the Maud Creek Gold Project. This report is based on information provided by Newmarket Gold to SRK and verified during site visits conducted in 2015 and any additional information provided by Newmarket Gold throughout the course of SRK' s investigations. The Qualified Persons have reviewed all relevant information and determined it to be adequate for the purposes of the Technical Report. The Qualified Persons do not disclaim any responsibility for this information. SRK has no reason to doubt the reliability of the information provided by Newmarket Gold.

This Technical Report is based on the following sources of information:

Discussions with Newmarket Gold personnel;

Inspection of Newmarket Gold' s Maud Creek Gold Project; and

Additional information and studies provided by Newmarket Gold.

2.4 Qualifications of SRK and SRK Team

The SRK Group comprises over 1,400 professionals, offering expertise in a wide range of Resource engineering disciplines. The SRK Group' s independence is ensured by the fact that it holds no equity in any project and that its ownership rests solely with its staff. This fact permits SRK to provide its clients with conflict-free and objective recommendations on crucial judgment issues. SRK has a demonstrated track record in undertaking independent assessments of Mineral Resources and Mineral Reserves, project evaluations and audits, technical reports and independent feasibility evaluations to bankable standards on behalf of exploration and mining companies and financial institutions worldwide. The SRK Group has also worked with a large number of major international mining companies and their projects, providing mining industry consultancy service inputs.

The compilation of this Technical Report was completed by Peter Fairfield, Principal Consultant (Project Evaluation), BEng (Mining), FAusIMM (No 106754) CP (Mining). By virtue of his education, membership to a recognised professional association and relevant work experience, Peter Fairfield is an independent Qualified Person (QP) as defined by NI 43-101.

Danny Kentwell, Principal Consultant (Resource Evaluation), MSc Mathematics and Planning (Geostatistics), FAusIMM, undertook a review of the Mineral Resources and geological aspects of the project and contributed to the relevant sections in this Technical Report. By virtue of his education, membership to a recognised professional association and relevant work experience, Danny Kentwell is an independent QP as defined by NI 43-101.

Simon Walsh, SRK Associate Principal Metallurgist, BSc (Extractive Metallurgy & Chemistry), MBA Hons, CP, MAusIMM, GAICD undertook a review of the metallurgical, mineral processing and infrastructure aspects of the project. By virtue of his education, membership to a recognised professional association and relevant work experience is an independent QP as this term is defined by NI 43-101.

Table 2-1: Site Visits

QP	Position	Employer	Last Site Visit Date	Purpose of Visit
Peter Fairfield	Mining Principal Consultant	SRK	13 Aug 2014	Site Inspection
Simon Walsh	Processing Principal Consultant	Simulus	13 Aug 2014	Site Inspection
Rodney Brown	Geology Principal Consultant	SRK	13 Aug 2014	Site Inspection
Louie Human	Geotechnical Principal Consultant	SRK	3-7 Aug 2015	Geotechnical Logging
Tristan Cook	Geotechnical Consultant	SRK	3-7 Aug 2015	Geotechnical Logging
Kirsty Sheerin	Geology Consultant	SRK	3-7 Aug 2015	Geological Logging

2.5 Acknowledgement

SRK would like to acknowledge the support and collaboration provided by Newmarket Gold personnel for this assignment. Their collaboration was greatly appreciated and instrumental to the success of this project.

2.6 Declaration

SRK' s opinion contained herein is based on information collected by SRK throughout the course of SRK' s investigations, which in turn reflect various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results may be significantly more or less favourable.

This report may include technical information that requires subsequent calculations to derive sub-totals, totals and weighted averages.

Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material.

SRK is not an insider, associate or an affiliate of Newmarket Gold, and neither SRK nor any affiliate has acted as advisor to Newmarket Gold, its subsidiaries or its affiliates in connection with this project. The results of the technical review by SRK are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings.

3 Reliance on Other Experts

This report has been prepared for Newmarket Gold and is based, in part, as specifically set forth below, on the review, analysis, interpretation and conclusions derived from information which has been provided or made available by Newmarket Gold, augmented by direct field examination and discussion with former employees, current employees of Newmarket Gold.

The Qualified persons have reviewed such technical information and determined it to be adequate for the purposes of this Technical Report. The Qualified Persons do not disclaim any responsibility for this information.

SRK has not performed any sampling or assaying, detailed geological mapping, excavated any trenches, drilled any holes or carried out any independent exploration work.

SRK did undertake geological and geotechnical logging of specific drillholes to assist in validating project assumptions.

4 Property Description and Location

The Maud Creek Deposit of Newmarket Gold described within this Technical Report is located within the Pine Creek region of the Northern Territory of Australia (Figure 4 1). There are other projects managed and owned by Newmarket Gold in the Northern Territory which are discussed further in Section 23, they include:

The Union Reefs Gold Project and Processing Plant (Figure 4 2), located approximately 170 km south southeast of Darwin accessible by the Stuart Highway, 18 km north northeast of the Township of Pine Creek.

The Pine Creek Gold Project;

The Burnside Gold & Base Metals Project:

The Moline Gold and Base Metals Project; and

The Yeuralba Gold and Base Metals Project.

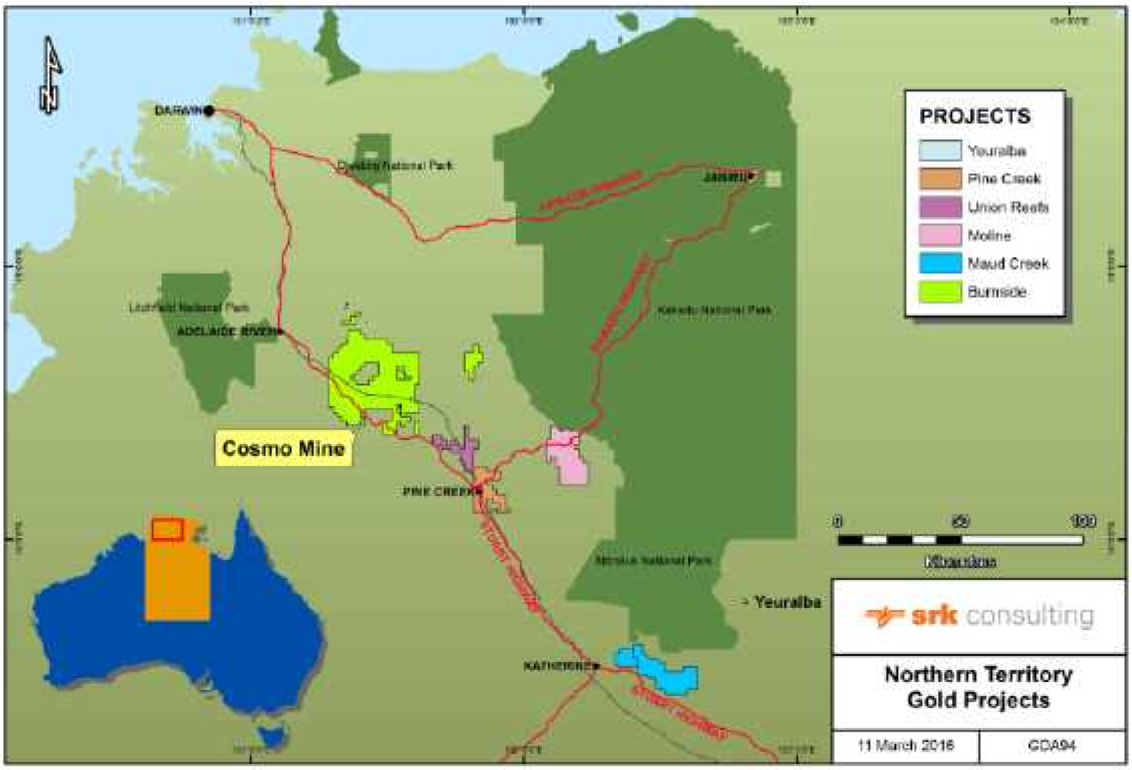


Figure 4-1: Newmarket Gold' s general location – Northern Territory gold properties

4.1 Property Location

The Maud Creek project comprises a total of 23 mineral titles (all granted) covering a total area of 29,489 ha (294.89 km2), as follows summarised in Figure 4-1.

Table 4-1: Summary of Mineral Titles Maud Creek Deposit

Licence Type	Number	Area (km2)
Exploration Licence		
Exploration Licence (EL)	2	280.26
Sub-total	2	280.26
Mineral Leases		
Mineral Lease (ML)	2	12.25
Sub-total	2	12.25
Total	**4**	**292.51**

The Project is located 20 km east of the regional administrative centre of Katherine (population 9,800) and is just east of the Township of Katherine. The deposit and proposed infrastructure is located on ML30260 which was granted 14 April 2014 and will expire on the 13 April 2024.

Geographically, the Project is centred about 6.7 km straight line distance northeast of the Stuart Highway, 287 km southeast of Darwin (population 129,100), the capital city of the Northern Territory (population 233,300), at Latitude 14°26' 41"S Longitude 132°27' 10" and UTM (AMG) coordinates (WGS 84, Zone 53L) 225407mE and 8401561mS, elevation 131 m ASL.

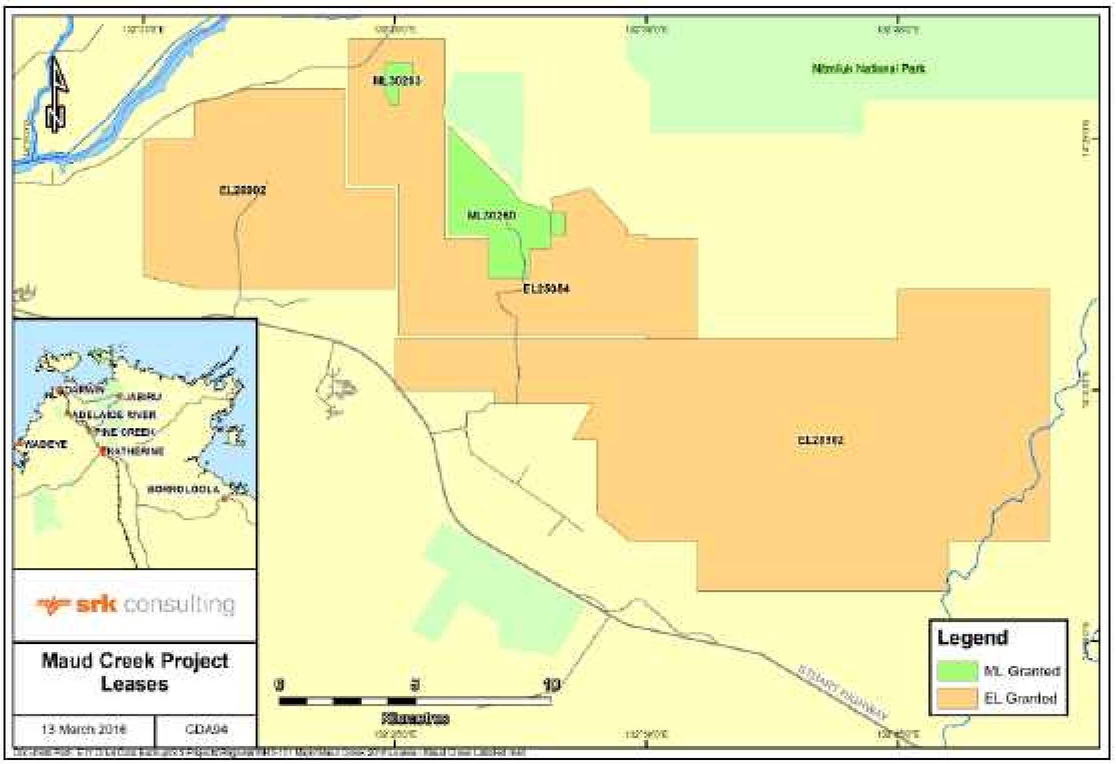


Figure 4-2: Maud Creek Gold Project Tenements

4.2 Land Tenure

The Maud Creek Project area lies within land traditionally owned by the Jawoyn people, who continue to exercise their traditional cultural attachment to the Katherine region as the owners and co-managers (with the NT Parks and Wildlife Commission) of the Nitmiluk National Park. The project lies on freehold land (NT Portion 4192, a subdivision of Portion 4159), outside lands administered by the Jawoyn Aboriginal Land Trust and the proposed mining operations area does not intersect any other Aboriginal land trust parcels.

The key mining tenement required for implementation of the Maud Creek project is ML 30260. Newmarket Gold was granted tenure over ML30260 on 14 April 2014. The current tenure expires on 13 April 2024, at which time the tenement holder has the option to renew its holdings, providing it has adhered to tenement conditions and to reporting and expenditure obligations.

There are no registered or determined native title claims over the Project area.

A land use agreement is in place for the titles shown in Figure 4-3. This is termed the Michell Compensation Agreement, which was original signed in 1992 between Michell, Biddlecombe and Trescabe to compensate the landholder for being deprived of the use of the surface of land, any damage to the property through exploration activities and being deprived of land improvements. This agreement has been assigned and accepted by Newmarket Gold.

There is an agreement between Newmarket Gold and the estate of Robert Biddlecombe relating to titles EL7775, EL8018 and MCN' s 4218 4225, (Figure 4-4), inclusive relating to a royalty payment for the mining of gold on these tenements. This is discussed further in Section 22.2.5.

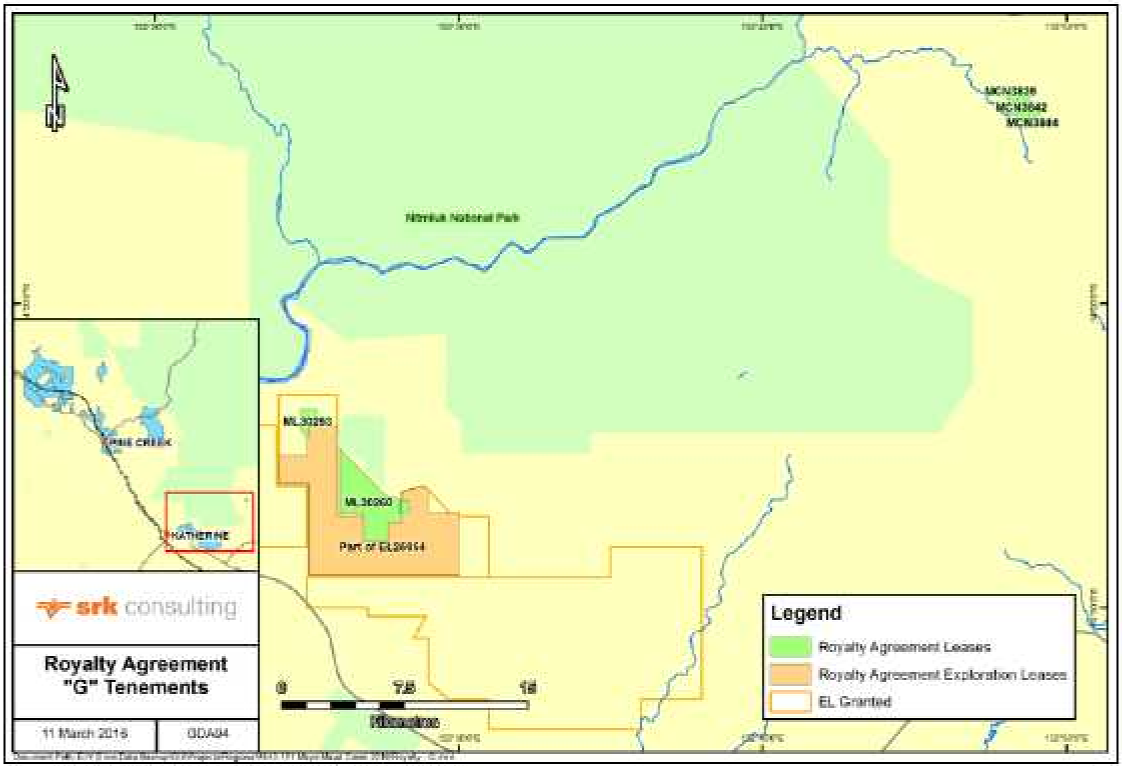


Figure 4-3: Agreements for Maud Creek Royalty

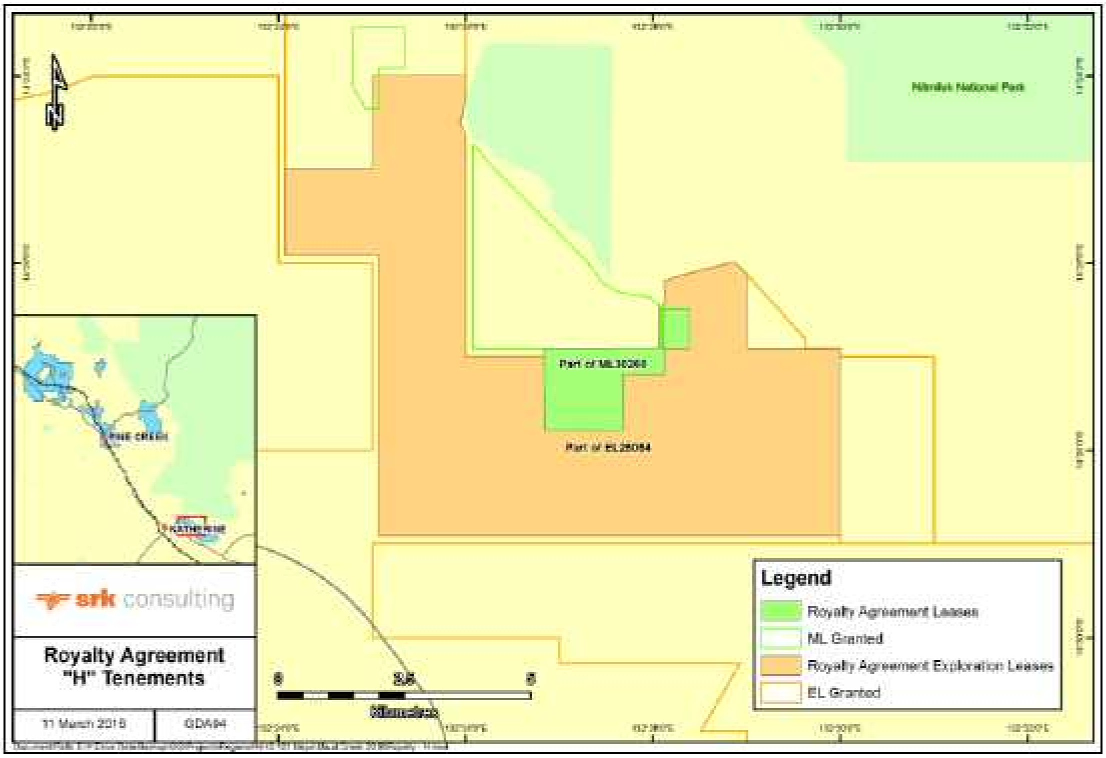


Figure 4-4: Agreements for Maud Creek Royalty with Biddlecombe

4.3 Underlying Agreements

4.3.1 Royalties

The following is a summary of the agreement and royalties, provided to SRK by Newmarket Gold, further detail is presented in Section 22.2.5.

Government Royalty – payable to the Northern Territory under the *Mineral Royalty Act (NT)*. The royalty rate is 20% of the net value of a saleable mineral commodity (in this case the gold concentrate) sold (or removed without sale) from a production unit (i.e. ML 30260) in a royalty year. The net value for the production is calculated by the following formula:

Net Value = Gross Realization – (operating costs + capital recognition deduction + eligible exploration expenditure + additional deductions)

Harmony Royalty – payable to Harmony Gold Operations Limited pursuant to a Deed of Assignment and Assumption dated 2 September 2009. Applies to all of ML 30260. The royalty rate is 1% of the value of all gold as defined in the agreement (i.e. the Perth Mint price for gold with no deductions). The royalty is not payable before 250,000 ounces of gold produced. Note also the Decision to Mine payment of AUD2M (indexed to CPI)

Virotec Royalty – payable to Mt Carrington Mines Pty Ltd pursuant to a Deed of Assignment and Assumption dated 2 September 2009. Applies only to that part of ML 30260 that was formerly within MCNs 4218 to 4225. Royalty rate is AUD5.00 per ounce with respect to 80% of the gold produced.

Biddlecombe Conglomerate Royalty – payable to Robert Biddlecombe (estate) pursuant to a Deed of Assignment and Assumption dated 2 September 2009. Applies only to that part of ML 30260 that was formerly within MCNs 4218 to 4225. Royalty rate is 1% of the gross value received as sale proceeds of all mineralization, metals, minerals and other products, after payment of the expenses incurred in smelting and refining charges.

Note that the above summaries are based on a plain reading of the documents.

NT Build Levy

Although it is not a royalty, for the purposes of the financial modelling of the Maud Creek project the construction works associated with the Maud Creek project may be subject to a levy under the *Construction Industry Long Service Leave and Benefits Act (NT)* (*Levy Act*).

In summary, the Levy Act imposes a levy on Construction Work in the Territory where the costs of construction work, commenced after 7 April 2014, are AUD1 million or more. The levy must be paid, prior to construction works commencing, by the person for whom the work is to be done (i.e. Newmarket Gold). The definition of Construction Work applies to civil works and works for buildings and structures that form part of the land, including a range of repair and maintenance works with respect to such civil works or buildings and structures.

The levy rate for construction works that commence after 7 April 2014 is 0.1% of the costs of the construction work. The Levy Act specifies that the costs of Construction Work is the total contract prices for all the construction contracts in relation to the work.

4.3.2 Farm-out Agreement

Farm-out agreements provide for third parties to explore on mineral titles, which are not owned 100% or substantially controlled by Newmarket Gold. The following discusses agreements relevant to the Maud Creek Project.

On November 6, 2013, Thundelarra Exploration Limited Uranium Exploration (Thundelarra) withdrew from a joint venture agreement with Newmarket Gold. Thundelarra was replaced by Rockland Resources Pty Ltd (Rockland) as party to the joint venture agreement, a 100% owned subsidiary of Oz Uranium Pty Ltd. Rockland was then replaced as a party to the agreement with Oz Uranium Exploration Agreement for the Pine Creek Tenements. Rockland Resources Pty Ltd (Rockland), a wholly-owned subsidiary of Oz Uranium, and Crocodile Gold formed a joint venture on November 6, 2013, in regards to uranium exploration and development on the Maud Creek, Burnside, Cosmo, Pine Creek, Union Reefs and Moline projects. Rockland has a minimum expenditure commitment of AUD1 million over the next four years. Rockland has the rights to apply for a mining tenement in its own right as long as it does not conflict with Newmarket Gold' s operations.

Over the past 24 months Rockland has been active in the Pine Creek region. They have conducted regional scale geophysical surveys and reviews (VTEM) as well as geochemical analysis, structural mapping and drilling in and around their currently identified uranium deposits. While one prospect is close to the Cosmo Mine (Fleur de Lys) no work has been conducted by Rockland on MLN993.

Land and Mining Property Swap Agreement - 2008

Land & Mining Property Exchange Deed (unregistered) and Land Use Deed dated 2008 (unregistered) Parties involved:

GBS Gold Australia (Land Holdings) Pty Ltd and Terra Gold Mining Pty Ltd (previously Terra Gold Mining Limited)

Teelow Nominees Pty Ltd and Michael Daniel Teelow.

BACKGROUND

This agreement relates to the transfer of the Moline Project from Michael Teelow to GBS Gold in exchange for the transfer of the Maud Creek farm and NT portion 4192 from GBS to Michael Teelow. The titles transferred to GBS included MLN 41 and 1059, EL' s 23605, 22966, 22967, 22968, 22970, 24262 and 24127 and MLA 24173.

PARTICULARS

As part of the property swap agreement Teelow the lease owner has the first right of refusal on any land sale at the Maud Creek farm. The tenement owner also has the Right of way over the property which allows the tenement owner the rights to establish easement over the farm. Newmarket Gold exercised this right and has established two easements over the farm for future access to the mine.

4.3.3 Farm-in` Agreement

In 2014, Phoenix Copper Pty Ltd (now PNX Metals) entered into a Farm-in agreement with Newmarket Gold. The Heads of Agreement was signed in August 2014 and was completed in December 2014. The Farm-in agreement relates to exploration activities on the Burnside Exploration Licenses as well as at the Chessman (close to Maud Creek) and Moline projects.

The Farm-in Tenements include the Maud Creek Project including exploration licenses EL25054 and EL28902, and mineral lease ML30293.

The PNX Metals agreement does not relate to ML30260. Newmarket Gold holds 100% rights to the Maud Creek Project and PNX have no interest.

PNX Metals has been active since signing the Heads of Agreement in August 2014.

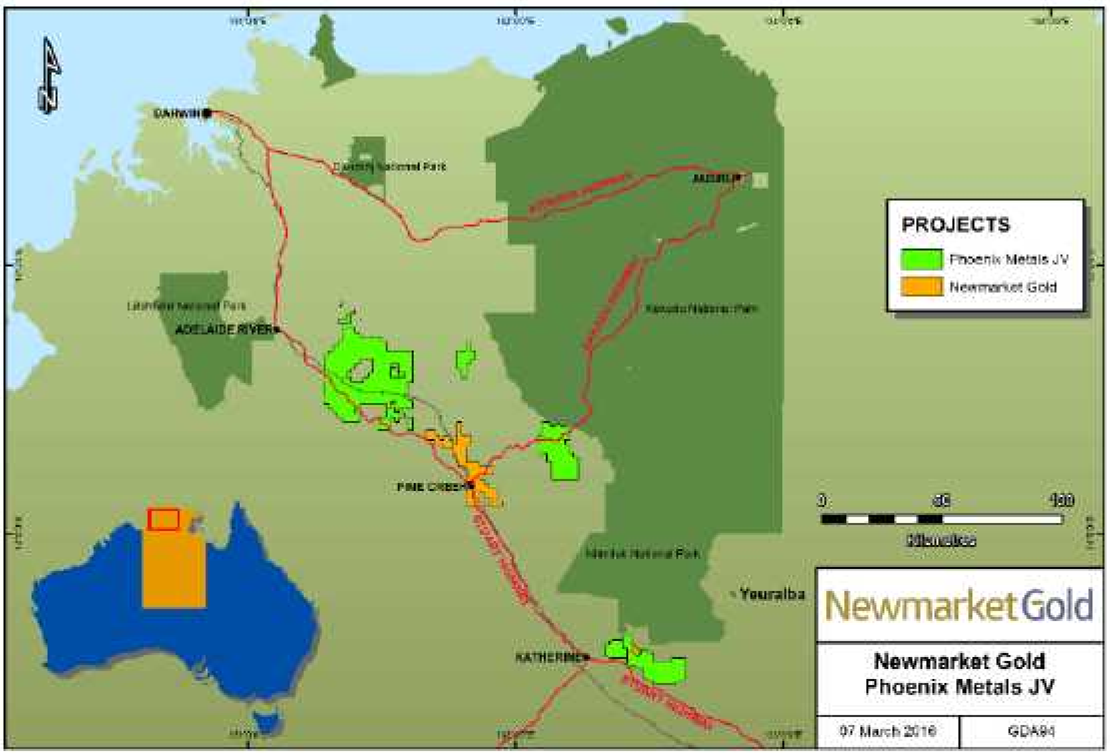


Figure 4-5: Agreements for Maud Creek Farm-in with PNX Metals

4.4 Environmental Liability

In addition to any environmental impacts that would arise in connection with new mining or mineral processing activities, Newmarket Gold would generally be liable for the management and eventual rehabilitation of legacy impacts present at the Maud Creek site at the time that Newmarket Gold took ownership of the Project.

Exploration for a range of commodities (copper, molybdenum, uranium and gold) has occurred intermittently at Maud Creek since approximately 1890. Small scale gold mining and a limited amount of processing is reported to have occurred at the site in 1890 - 1891 and again in 1932 - 1934 (Mining One, 2013).

The most recent mining at Maud Creek occurred in 2000, when Katherine Mining NL conducted open cut mining for gold. Ore was treated offsite. In the order of 9 ha of disturbed land (comprising 2.7 ha associated with the pit void, 1.6 ha associated with the former ROM pad and approximately 4.7 ha occupied by a waste rock dump) remain from previous mining. Minor disturbance related to support infrastructure (access tracks, relocatable offices) also remains (Figure 4-5).

Vegetation mapping conducted in 2007 as part of baseline environmental studies for an environmental impact assessment of the proposed Terra Gold mining project at Maud Creek mapped an area of approximately 14 ha as cleared for mining and the Terra Gold EIA reported that Approximately 96 ha of savannah woodland vegetation has been cleared in the Maud Creek project area for pastoral development and to support exploration and historical mining activity. (Crawford and Metcalfe, 2007; URS, 2008). Exploration disturbance (drillholes) arising during exploration activities subsequent to Crocodile Gold' s acquisition of the Maud Creek tenements in 2009 is reported to have been rehabilitated (Mining One, 2013).

In addition to direct clearing, historic mining activity is likely to have contributed to the establishment and spread of weeds, which are reported to be abundant and well established at Maud Creek, with dense infestations, especially along access roads and drainage lines and in disturbed areas. Some of the weeds recorded in the Project area are declared weeds under the NT *Weed Management Act*. Landholders are required to make a reasonable attempt to control and prevent the spread of declared weed species (Department of Land Resource Management, 2015).

Legacy features from previous mining at Maud Creek (notably the waste rock dump and pit void) may represent a potential source of acid or metalliferous drainage, however the limited surface and groundwater quality data for the site do not so far show the presence of significant impacts: water quality downstream of the site and in water storages at the site is generally in the range of values recorded upstream.

Figure 4-6: Existing disturbance at Maud Creek (May 2007) (from URS, 2008)

Mining activities in the Northern Territory are fully bonded. That is, the NT government requires lodgement of a security to cover 100% of the estimated cost of rehabilitating any disturbance proposed by the tenement holder under its Mining Management Plan (MMP), together with the cost of rehabilitating any pre-existing disturbance on the tenement. Additionally, from 2013, tenement holders are required to make annual contributions to the Mine Rehabilitation Fund (MRF) established under the *Mining Management Act*. The MRF payments are a non-refundable annual levy of 1% of the total calculated rehabilitation cost applied to each mining operation.

SRK has not independently estimated the cost of rehabilitating existing disturbance within the Maud Creek tenements as part of this initial review.

Permitting of the Project would require the preparation of a Mining Management Plan (MMP), including a cost estimation of mine rehabilitation and closure works. The NT Government has developed an Excel spreadsheet for estimating the security deposit to be lodged to cover mine rehabilitation and closure.

4.5 Legislation and Permitting

In the Northern Territory environmental impact assessment and subsequent authorisation and regulation of the implementation of mining and related support activities is chiefly administered until three Acts:

The Environmental Assessment Act 1982,

The Mining Management Act 2001, and

The Waste Management and Pollution Control Act 2009.

The *Mineral Titles Act* 2010 also exerts a considerable influence on regulation of environmental aspects of mining activities in that it provides for a number of significant exemptions to licensing provisions under other Acts that would otherwise apply.

If a mining proposal has the potential to give rise to significant adverse impacts on a 'Matter of National Environmental Significance then it may also require referral to and assessment by the federal Department of the Environment (DotE) under the Environment Protection and Biodiversity Conservation Act 1999 (EPBC Act). A simplified flow chart showing the environmental impact assessment process is presented in Figure 4-6.

Although there are some differences in the duration and fine detail of the Public Environmental Review (PER) and Environmental Impact Statement (EIS) pathways, the overall processes are similar. The red arrow on the figure shows the point at which Terra Gold' s environmental permitting was terminated by the proponent.

The Maud Creek Project currently contemplated by Newmarket Gold has not been referred to the Northern Territory Environmental Protection Authority (NTEPA) or DotE for assessment. In 2006 Terra Gold referred a proposal for mining and processing of mineralization from Maud Creek to the NTEPA for assessment. The EPA determined that Terra Gold' s Maud Creek proposal should be assessed via the EIS pathway. A draft EIS report was issued for public review and comment in early 2008, but the proposal was ultimately withdrawn without completing the assessment process. Terra Gold also referred its Maud Creek proposal to the Commonwealth in 2006. In early 2007, the federal government determined that the Project was not a 'controlled action' under the EPBC Act.

It is likely that any future assessment of the Maud Creek project would follow a similar assessment path. Although much of the information produced for Terra Gold' s EIS would still be relevant, it would almost certainly be necessary to recommence the Project' s environmental assessment from the NOI stage (i.e., it would not be possible to re-activate the project assessment starting at the public exhibition phase). There is no fixed statutory timeline for the EIS process. A minimum of 18 to 24 months is typically required to progress from the NOI stage to completion of the EPA assessment.

Once the EIS process has been completed, permitting of operational aspects of the Project would be administered primarily under the *Mining Management Act* and the *Waste Management and Pollution Control Act.* Apart from the approvals required under these Acts, a range of authorisations or implementation conditions may arise under the following legislation:

Water Act 1992

Heritage Act 2011

Northern Territory Aboriginal Sacred Sites Act 1989

Planning Act 1999

Dangerous Good Act 1998

Transport of Dangerous Goods by Road and Rail (National Uniform Legislation) Act 2011

Territory Parks and Wildlife Conservation Act 2000

Native Title Act 1993 (Cth)

Aboriginal Land Rights (Northern Territory) Act 1976 (Cth)

Additional information on permitting of specific environmental and heritage aspects of the Project is presented in Section 20.2

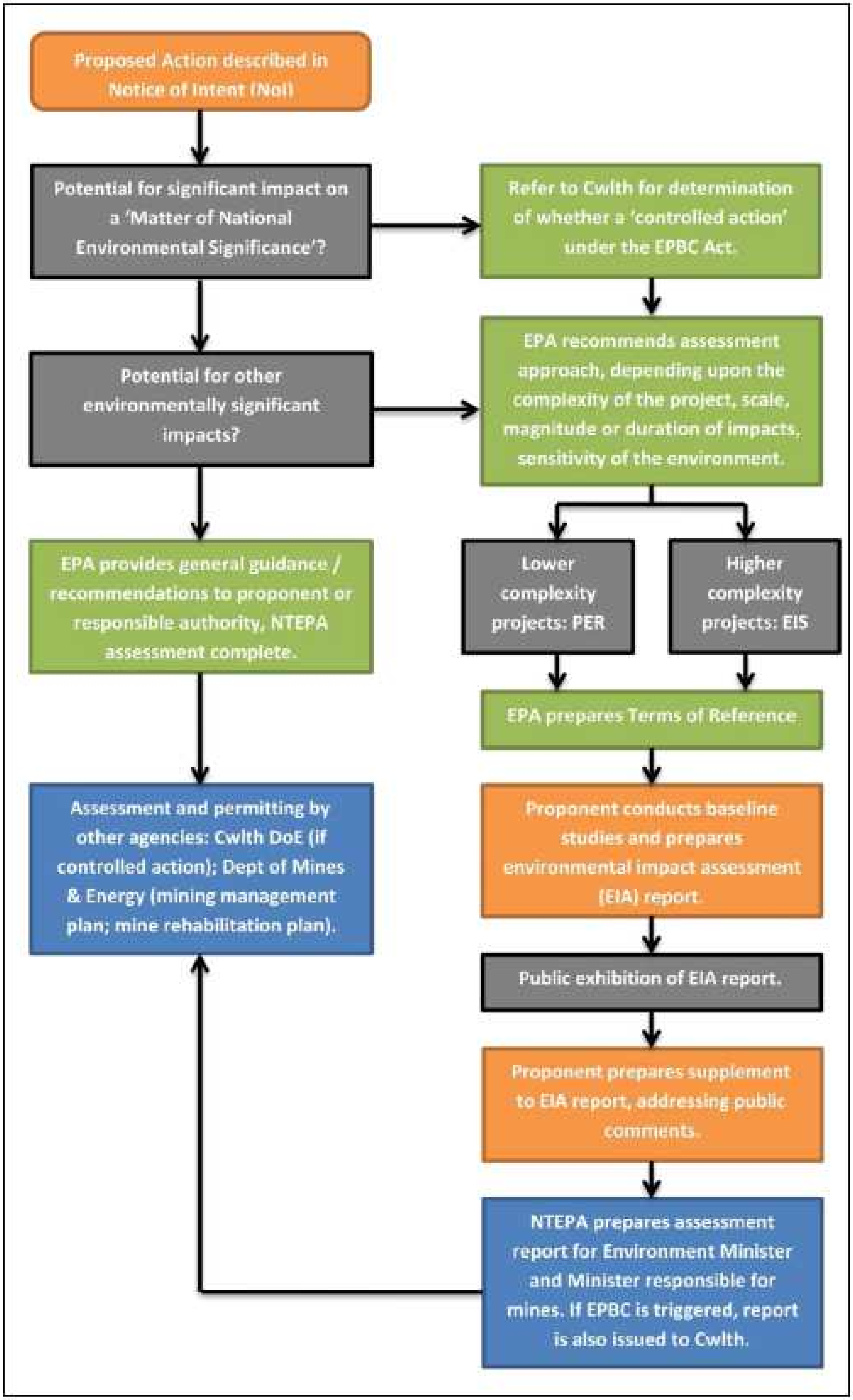


Figure 4-7: Simplified process diagram – NT environmental assessments

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Northern Territory is the least populated of all areas in Australia. It encapsulates a total area of 1.35 million square kilometers and accounts for 20% of the whole country; however, just 233,300 (ABS March 2012) or 1% of Australia's population reside there. The Territory varies considerably in topography, climate, and infrastructure. The region is dry between April and September, and wet between October and March. During the wet season everything is green and there is no dust; however, the humidity and temperatures are high and access off road is difficult. The centre is extremely arid, with greatly varying temperatures and is known as the Red Centre named because red is the predominant color found in the soil.

Darwin, Capital of the Northern Territory, lies on the coast to the north and provides the majority of infrastructure support and services for the mining industry. The Stuart Highway, which virtually bisects the country, is the main road that leads from Darwin to Alice Springs then on to Adelaide in South Australia.

5.1 Accessibility

Access is gained to the Project from Darwin by travelling south for some 314 road kilometers along the sealed Stuart Highway to the town of Katherine.

The Stuart Highway, the area' s major thoroughfare, and the Adelaide to Darwin transcontinental railway line bisect Australia in a north south sense and provide access to the Maud Creek Property. The Project site is approximately 30km from the town of Katherine.

The Union Reefs Processing Plant, owned and operated by Newmarket Gold is located approximately 185 km southeast of Darwin, 15 km north of the town of Pine Creek.

5.2 Land Use

Major land uses are traditional Indigenous uses, nature conservation (including parts of Kakadu National Park and World Heritage Area and Litchfield National Park), urban and other intensive uses and grazing. Approximately 85,000 hectares have been cleared. The region has undergone some localized clearing and the major land uses are grazing, nature conservation (including parts of Kakadu National Park and World Heritage Area and Litchfield National Park), traditional Indigenous uses and other intensive uses including horticulture.

The Daly Basin Bioregion consists of gently undulating plains and scattered low plateau remnants and has a tropical monsoonal climate with distinct wet and dry seasons and high temperatures throughout the year. Dominant vegetation is tropical eucalypt woodlands/grasslands and eucalypt open forests. Smaller patches of eucalypt woodlands and melaleuca forests and woodlands are present.

The major land use is grazing on native pastures and traditional Indigenous uses with some horticulture, grazing on modified pastures and nature conservation. The region has undergone some clearing (approximately 167,000 ha) for these developments The vegetation of the Maud Creek area consists largely of woodlands and open woodlands (predominant species – Eucalypts) that have been degraded by the impacts of cattle, buffalo and wild donkeys. No rare, threatened or endangered species have been identified in the area.

5.3 Topography

Generally the topography of the Property area is flat, locally gently undulating.

In the Maud Creek area, the terrain is flat lying to undulating. Ephemeral streams transect the project area and drain into the westward flowing Katherine River that flows all year. Land units occurring within the Maud Creek area include:

Rugged terrain with slopes 15 to 40% with shallow or skeletal soils;

Hilly terrain with slopes 5 to 15%, rocky and boulder strewn with shallow and skeletal soils;

Gently undulating crests and upper slopes to 5% with shallow rocky soils;

Undulating terrain with slopes 5 to 10% with grey and brown clays; and

Major creeks and gullied tributaries.

5.4 Climate

The Top End of the Northern Territory has a tropical monsoon climate characterized by two distinct seasonal patterns: the 'wet' monsoon and the 'dry' seasons. The wet season generally occurs from November through to April and the dry season between May and October. Almost all rainfall occurs during the wet season, mostly between December and March, and the total rainfall decreases with distance from the coast.

The mean daily maximum temperature, as recorded at Darwin on the northern coastline, is 31°C in the coolest months of June to August and 33°C in the hottest months of October and November. The mean daily minimum temperature in Darwin range from approximately 19°C (dry season) to 25°C (wet season). The average annual rainfall at Darwin is 1,713 mm.

The mean daily maximum temperature, as recorded at Katherine, is 31°C in the coolest months of June to August and 38°C in the hottest months of October and November. The mean daily minimum temperatures at Katherine range from approximately 13°C (dry season) to 24°C (wet season). The average annual rainfall at Katherine is 971 mm.

During the wet season, high intensity rainfall events are common, resulting in local flash flooding of ephemeral streams and watercourses. Mining operations are continuous throughout the year; however, increased stockpiling is undertaken in the lead up to the wet season thereby offsetting the reduced mining movements over that period. Experience has shown that it is best to shut down hauling during periods of extreme rainfall as damage to haul roads by large trucks may occur quickly.

The annual evaporation rate remains high throughout most of the Northern Territory, ranging from 2,400 mm to 4,000 mm per annum. Monthly evaporation exceeds rainfall for eight months of the year at the coast increasing to the whole year inland. It remains relatively high even during the wet season.

Climate gradually moves from seasonally wet tropical in the north to arid in the south, with corresponding changes in landscape, with areas of rocky escarpment and plateau which break a low relief in the north and rocky ridges in the south.

The Northern Territory has a diversity of vegetation that is maintained by its variety of climate and soils. Natural vegetation of the Properties is typical of savannahs of the northern part of Australia, dominated by Eucalypt species with a grassy understorey dominated by sorghum species. The Northern Territory is the only area in Australia that does not have conspicuous temperate flora.

In the north, the vegetation is typically tropical savannah (eucalypt woodland and eucalypt open woodland with a grassy understory). This landscape experiences dramatic seasonal changes with intense growth in the wet season (summer) and widespread fires in the dry season (winter). Famous worldwide for the tropical wetlands and rugged sandstone escarpments of Kakadu National Park, the wetlands are of importance for conservation, providing breeding areas, habitat and refuge for important wildlife populations.

From the north, a transition area moves from eucalypt woodlands into areas of melaleuca and acacia forests and woodlands and south into the spinifex (hummock grasslands), Mitchell grass (tussock grasslands) and acacia woodlands and shrublands. The vegetation increases in diversity around Alice Springs with areas of mulga, mallee, chenopods, hummock grasslands, small pockets of eucalypt woodlands and salt lakes.

5.5 Infrastructure and Local Resources

Darwin has a population in excess of 129,000 and is the capital city of the Northern Territory. It is the administrative centre of the Northern Territory government and a major transportation hub, with an international airport and deep water port and the Adelaide to Darwin transcontinental railway terminating at the East Arm port. As the largest city in the Northern Territory, Darwin also has excellent schools, hospitals, and retail, commercial and light industrial services.

A considerable proportion of consumer and other goods reaching the Northern Territory are brought by road from Queensland or South Australia. The Stuart, Arnhem, Kakadu, Barkley and Victoria Highways ensure high service levels to the Darwin region from the Australian capitals and other regional centres.

Despite its low population, the area between Darwin and Katherine in the Northern Territory is well serviced with infrastructure. Significant mining operations have been developed in the area over the past 30 years, with gold mining and processing operations conducted within or in close proximity to the project areas at Cosmo Howley, Brocks Creek, Pine Creek, Mount Todd and Union Reefs.

Katherine is a regional centre with a population of approximately 9,800 and enjoys excellent infrastructure, services and communications. This is the closest centre of population to the Maud Creek project.

The regional mining communities of Pine Creek (with a population of 450) and Adelaide River (population of 200) support the Burnside, Maud Creek and Moline gold projects.

The Arnhem Highway to the east southeast of Darwin provides a communication link to the Kakadu National Park and Jabiru, a town of 1,135, which provides accommodation for the uranium mines in the vicinity. Accommodation and services are available along the highway, primarily for the tourist trade.

6 History

6.1 Introduction

Gold was initially discovered in the Maud Creek area in 1890 and a small plant was set up but ultimately abandoned in 1891. This is now called the Chlorite Hills and O' Shea' s area.

The area was re looked at from 1932 34 when 400 tonnes of ore produced 540 ounces of gold. Mining was from about 20 shallow shafts and small holes that were 6 12 m deep with horizontal workings from 15 30 m in length in the Chlorite Hills and O' Shea' s area.

Interest in the Maud Creek area was rekindled in the 1960s during an assessment of the mineral potential of the Top End of the Northern Territory. This study was prompted by the discovery of significant uranium mineralization in the nearby South Alligator River valley in the mid 1950s.

The Northern Territory Geological Survey carried out IP surveys, soil sampling and petrographic investigations in the late 1970s as part of an assessment of an extension to the nearby township of Katherine.

6.2 Ownership and Exploration Work

Between 1966 and 1973 several companies including Western Nuclear Australia and Magnum Exploration NL explored the area for copper, gold and uranium. IP surveys and drilling of siliceous and gossanous breccias intersected low, albeit anomalous, concentrations of copper and molybdenum and numerous pyritic zones.

In 1973 Magnum Exploration NL (EL147) explored the breccia in the Red Queen/Chessmen area as part of a copper uranium search. They considered the breccia to be similar to the Rum Jungle occurrence. They drilled 7 holes into the breccia and met with pyritic material with low copper values. They also dug trenches, and obtained anomalous copper and molybdenum values. They did not assay for gold.

In 1985 the Minerals and Exploration and Development Group (MEDG) of CSR Ltd explored the Maud Creek area. Stream sediment sampling returned a 1.3 ppb BLEG gold result about 1.3 km to the west of the old 19th century workings on Maud Creek (now called Chlorite Hills and O' Shea' s).

Placer Exploration purchased MEDG in 1987 and followed up on the BLEG anomaly with rock chip sampling and drilling which subsequently resulted in the discovery of the Gold Creek Zone.

Placer sold the deposit in 1992 to Kalmet Resources NL.

CSR Limited in 1986 held the Peckham Hill EL4874 that covered the Chessmen/ Red Queen prospect (located to the northwest of the Gold Creek Zone). They recognized that the breccia in the area previously mapped by the Bureau of Mines displayed epithermal textures. They conducted exploration programs comprising rock chip sampling, soil sampling, trenching and drilling at Red Queen and along strike. Some 10 km of strike of the breccia/veins were rock chipped and anomalous gold in a gossan sample was reported at Red Queen up to 6.63 g/t Au and 1.02% arsenic (AMG 8406034mN 221372E). The samples were also anomalous in copper, antimony, mercury and thallium.

Another gold anomalous breccia was located 1 km NE of Red Queen and this assayed 0.23 g/t Au. The remainder of the siliceous breccia gave values below 0.07 g/t Au. The north Chessmen location gave an anomalous stream sediment value (14.3 ppb gold at AMG 8406800N 331300E).

Soil sampling over the Red Queen prospect produced a peak value of 0.33 g/t Au at the western edge of the volcanic sediment contact. The anomalous breccia also gave positive soil values. Another zone was detected in the north with a value of 0.32 g/t Au in sediments, but without obvious structural association.

Trenching by CSR was carried out in the Red Queen area to test the soil anomalies numbered T1 to T7. T1 coincided with a Magnum Exploration (1973) trench and anomalous gold, arsenic and copper were reported with the best gold value at 6.72 g/t Au from quartz veins in mafic fragmentals. T2, also in an adjacent Magnum trench gave a peak value of 0.61 g/t Au. Trench T3, located 30 m to the SW, met with a carbonated zone with a maximum value of 1 m @ 0.42 g/t Au. T4, on a soil anomaly, met with 2 m @ 0.33 g/t Au. T5 and T6 did not explain the soil anomaly. T7 met with a best result of 2 m @ 0.46 g/t Au.

Percussion drilling was carried out totalling 1,210 m in 9 holes (CMPDH series). While promising mafic lithologies and chalcedony quartz carbonate alteration were met with, the results were sub economic with the best values falling in the range 0.1 to 0.4 g/t Au over intervals up to 19 m.

Between 1989 1990 Placer re established the CSR grid at Red Queen/Chessmen and conducted soil sampling (412 samples) on a 25 m grid. Samples were assayed for gold, copper, lead, zinc and arsenic. They reported two zones of gold anomalism, one over quartz veins and sheared chert and the other with the western contact zone. The anomalies displayed little correlation with the CSR soil anomalies. Elevated base metal values appeared to correlate with the hematite stained volcanics.

Rock chip sampling and mapping comprised 29 samples and assays up to 4.1 g/t Au were obtained from hematite float in the western contact zone.

Two lines of IP, dipole dipole, were carried out. A chargeability anomaly was attributed to black cherts. Ground magnetics were carried out over two airborne magnetic anomalies.

Five RC holes were drilled at Red Queen for 576 m. (RQP) Hole RQP5 drilled under a soil anomaly near the IP chargeability anomaly met with 10 m @ 0.95 g/t Au from 46 m and 8 m @ 0.97 g/t Au from 60 m. The associated lithology was black cherts. In general, attempts to correlate surface geology with the drilling proved confusing.

The Chessmen/Red Queen area was reduced to three MCNs and held as a second priority resource area following the discovery of the Main Zone gold deposit at Gold Creek. The trenches and drillholes were rehabilitated.

The Maud Creek project was owned by a number of companies until the acquisition of the project by GBS Gold in December 2006. Substantial drilling, in the order of 66,000 90,000 m of RC and diamond drilling, is reported on the project area during the period 1966 - 2006, oriented toward gold exploration.

During 1985 and 1986, CSR Limited (CSR) explored the area in an attempt to locate gold mineralization in the Lower Proterozoic dolerites. Work included 25 metre line spacing airborne magnetic and radiometric surveys, stream sediment sampling, soil sampling, rock chip sampling, petrographic sampling and trenching.

Between 1993 and 1997, Kalmet Resources NL (Kalmet) completed a series of drilling programs at Maud Creek. Metallurgical testing of five high grade RC samples was completed and an environmental impact study was commissioned. Metallurgical studies showed the primary gold bearing sulphide mineralization was refractory in nature and bio oxidation tests were initiated. A close line spaced airborne magnetic and radiometric survey was contracted over a significant part of the land position (Figure 6-1). Interpretation of this radiometric data was completed by Independent Engineers in 2005 (Figure 6-2 and Figure 6-3).

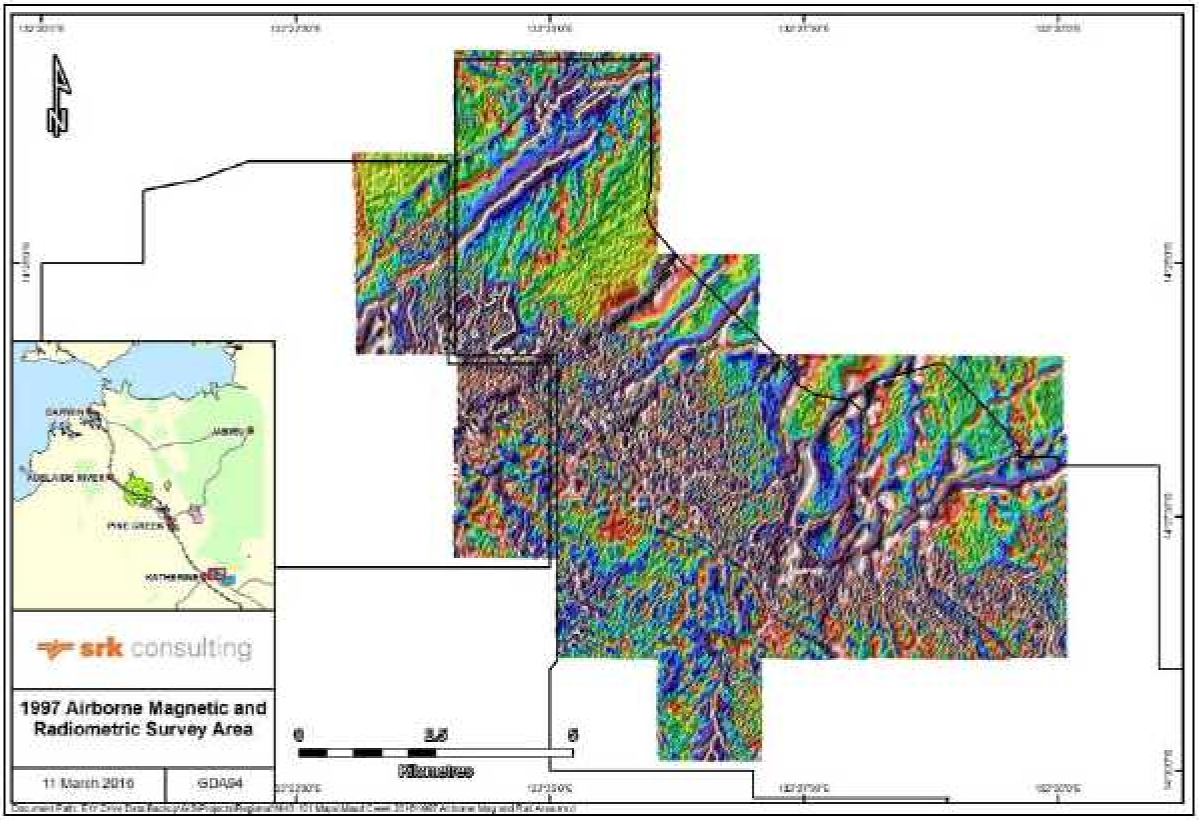


Figure 6-1: 1997 airborne magnetic and radiometric survey

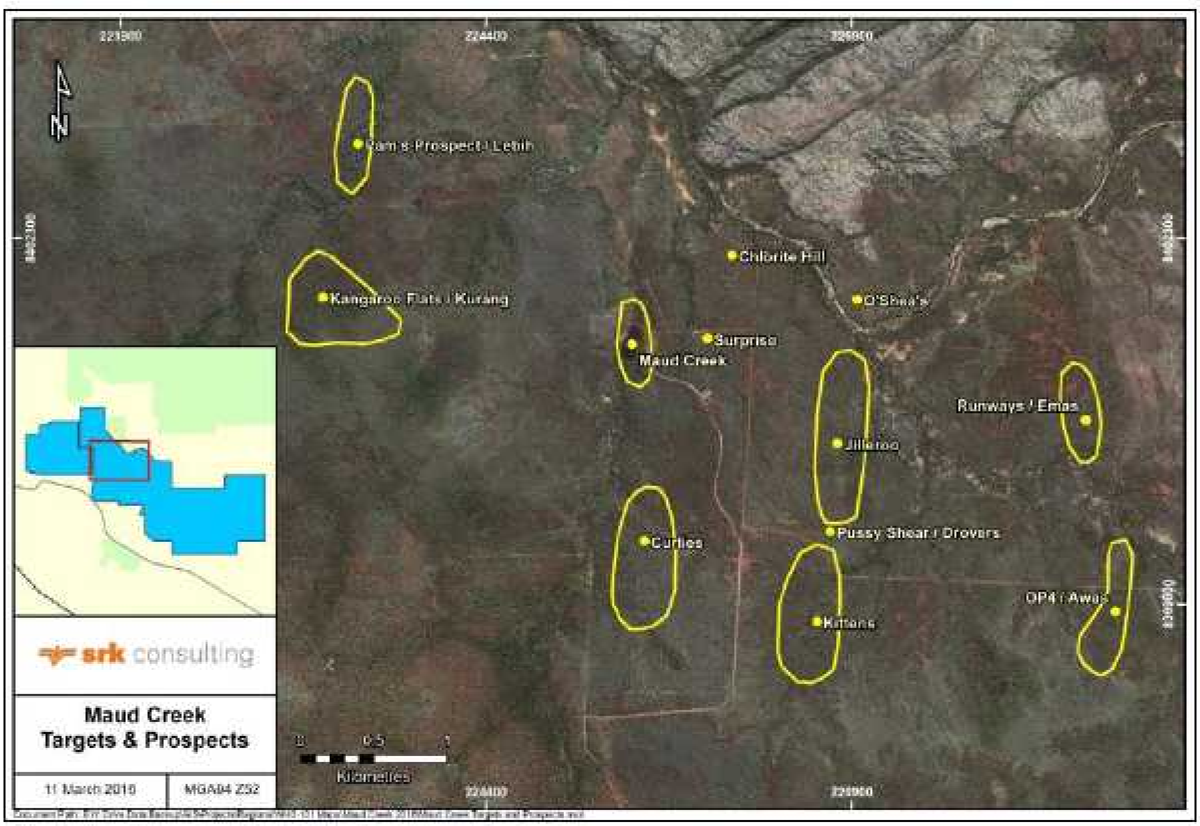


Figure 6-2: Maud Creek area pits and prospects (Independent Engineers)

Note: interpreted from radiometric data

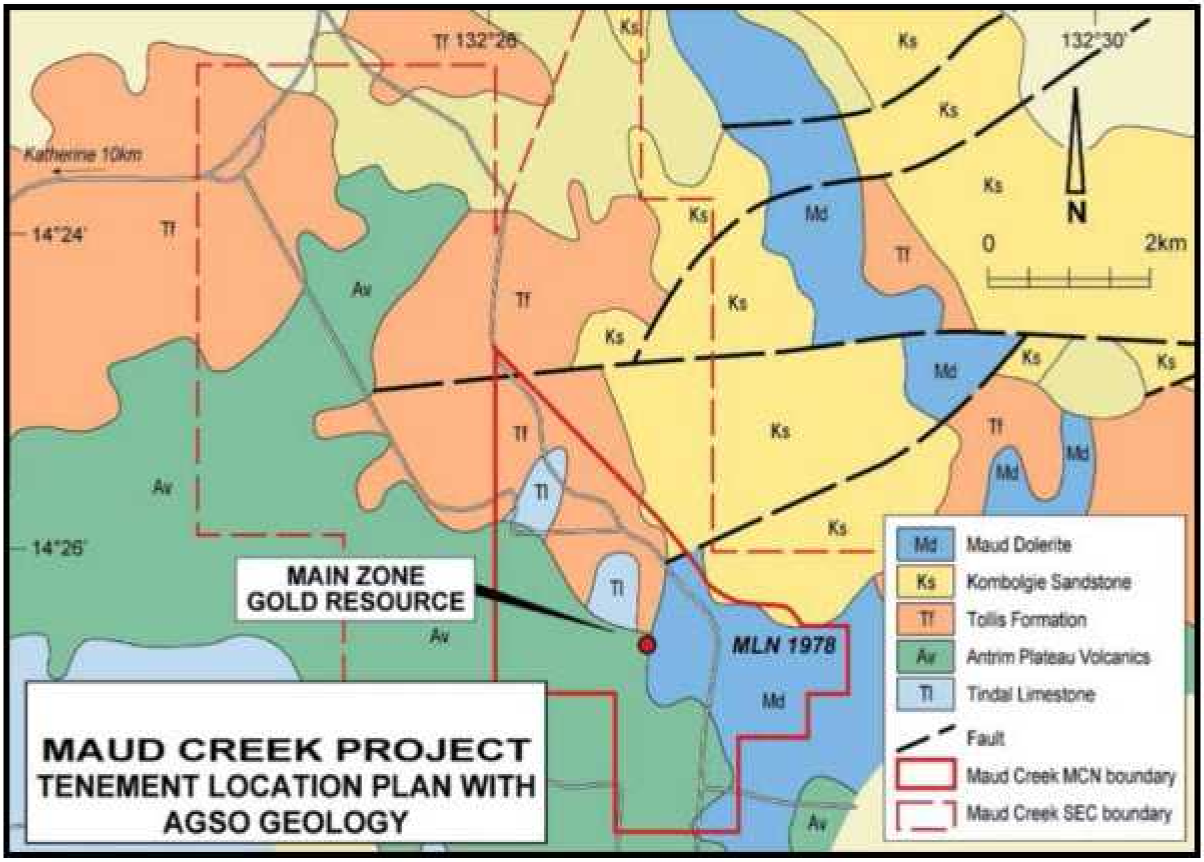


Figure 6-3: Maud Creek regional geology and structural interpretation by Independent Engineers 2005

In 1997, Kilkenny Gold NL acquired Kalmet and undertook RC and diamond drilling. They carried out significant drilling and increased the global resource to 995,000 ounces (Indicated and Inferred Mineral Resources).

Further metallurgical test work was completed including pilot scale flotation and bio oxidation program. In 1998, Signet Engineering completed a full feasibility study for the extraction and processing of oxide, transition and primary mineralization from Maud Creek. A comprehensive draft Environmental Impact Study was also produced.

In 1998 Kilkenny Gold commissioned SRK to complete a structural assessment and interpretation of aeromagnetic results. The report, maps and interpretation are quite detailed.

A major flood in the Katherine area in 1998 saw the loss of a significant amount of technical data in the form of reports and diagrams.

AngloGold acquired rights to mine the oxide zone of the Main Zone deposit at Maud Creek and treat the ore at the Union Reefs plant. Mining operations were conducted during 2000. A total of 173,581 tonnes grading 3.32 g/t Au for 18,527 ounces were obtained. Ore was trucked from Maud Creek to the Union Reefs mill.

Hill 50 Gold NL acquired the Maud Creek project from Phoenix Mining Ltd in March 2001 and conducted an extensive review of previous exploration, which identified five gold targets within the property. A program of rock chip sampling was conducted at the Runways prospect. Additional RC and diamond drilling was completed at Gold Creek and surrounding prospects.

In late 2001, Harmony Gold Company Ltd launched a takeover bid for Hill 50 and by mid 2002 had successfully completed the acquisition of the company and all its assets including the Maud Creek deposit. A photo geological interpretation of the property was completed by Snodin (2002), who recommend areas for further follow up work.

In December, 2004, and prior to being acquired by GBS Australia, Terra Gold Mining Pty Ltd (TGM) purchased an option to acquire the Maud Creek project from Hill 50, which by that time had been acquired by Harmony Gold. In January 2005, TGM drilled a single combined percussion diamond hole into the mineralized Main Zone to supply a limited quantity of sample for metallurgical test work purposes. Following a preliminary due diligence examination, in May 2005 TGM exercised its option to purchase the Maud Creek project.

In 2005, four holes totalling 711 m consisting of 406 m of RC pre collar and 305 m of HQ3 diamond drill core were completed. These holes were designed primarily to provide additional samples for metallurgical test work.

In August 2005, GBS Gold Australia Pty Ltd, a wholly owned subsidiary of GBS Gold, announced its intention to acquire all of the issued share capital of Terra Gold Mining, including its interest in the Maud Creek Gold Project. The acquisition was completed in January 2006 and the 100% interest was transferred to GBS Gold.

GBS completed resource calculations as well as mining, geotechnical and hydrogeological studies of the Maud Creek deposit. They also completed an extensive EIS report on the deposit area.

An agreement to acquire a number of properties, including the Maud Creek property, was entered into on June 19, 2009 from GBS Gold International Inc. (GBS Gold) (In liquidation). GBS Gold operated the Tom' s Gully and Brock' s Creek underground gold mines, mined several open pit gold deposits and operated two gold Processing Plants, one at Tom' s Gully, the other at Union Reefs, near Pine Creek, Northern Territory, until September, 2008, when administrators were appointed.

On November 6, 2009, the mining tenements including the Maud Creek Property were registered in the name of Crocodile Gold, which became Newmarket Gold in July 2015.

7 Geological Setting and Mineralization

7.1 Regional Geology

The Maud Creek Project lies within the Archean to Paleoproterozoic Pine Creek Orogen (PCO) which is located in the north of the Northern Territory and extends from Katherine in the south to Darwin in the north (Figure 7-1). The PCO is exposed over 47,500km^2 and consists of a deformed and metamorphosed sedimentary basin with a thickness of over 4 km and overlies a Neoarchean (ca 2670-2500 Ma) granitic and gneissic basement (Ahmad & Hollis, 2013). The PCO hosts over a thousand mineral occurrences and is recognised as one of the most prospective mineral provinces within Australia (Ahmad & Hollis, 2013). Known resources include uranium, gold, and platinum group metals (PGMs), as well as substantial base metals, silver, iron and tin-tantalum mineralization.

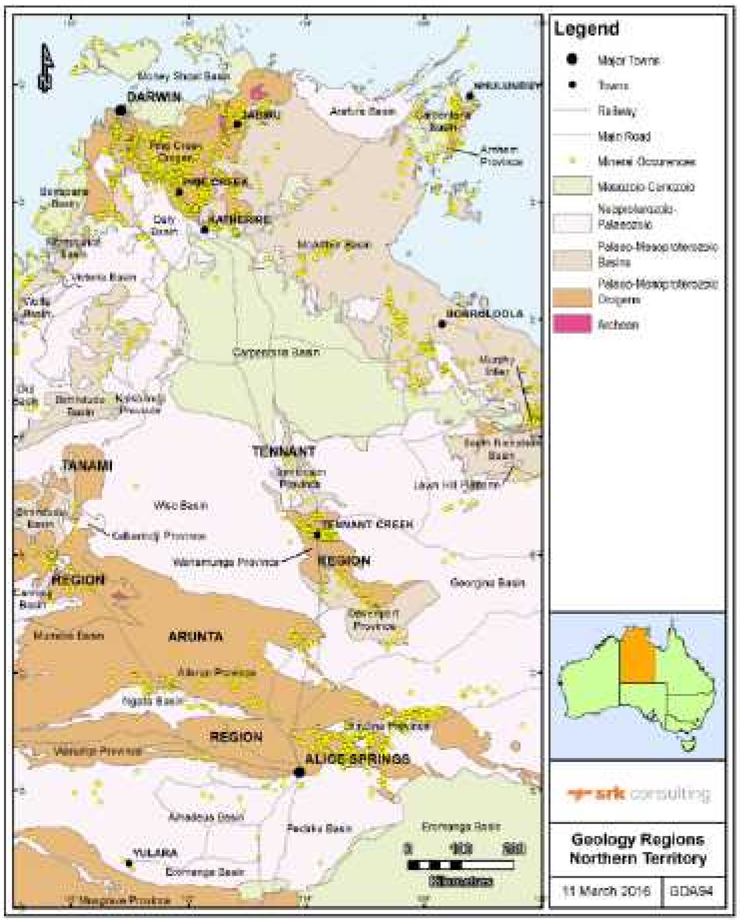


Figure 7-1: Pine Creek Orogen (northern orange zone) within the Northern Territory

Source: Ahmad & Hollis, 2013

The basement terrain of the PCO consists of a series of late Archean granite-gneiss basement domes which have subsequently been overlain by fluvial to marine sedimentary sequences of the Paleoproterozoic. These sequences have been divided into the Woodcutters and Cosmo Supergroups, which are separated by a major unconformity representing a time break of 160 Ma (Ahmad and McCready, 2001). Several highly reactive rock units are included within Cosmo Supergroup including carbonaceous shale, iron stones, evaporite, carbonate and mafic to felsic volcanic units of the South Alligator and Finniss River Groups. A northwest trending fabric is evident throughout these sequences resulting from greenschist facies metamorphism and multiphase deformation. A period of widespread felsic volcanism in aureoles between 500 m and 2 km wide overprint the earlier regional metamorphism following deposition of the Cosmos Supergroup (Snowden Report, 2008). A period of extension deformation following intrusions of these granitoids resulted in an extensive array of northeast and northwest trending dolerite dykes intruding the metasedimentary sequences.

Gold mineralization within the PCO is defined as orogenic in nature and are recognised to have common geological, geochemical, mineralogical and thermochemical characteristics (Ahmad & Hollis, 2013). Gold mineralization is commonly strongly structurally controlled within the region with gold exploiting structures such as anticlines, strike slip shear zones and duplex thrusts as well as located in proximity to the Cullen Granite Batholith (Snowden Report, 2008). Mineralization is commonly recognised within the upper Woodcutters Supergroup and Cosmos Supergroup, specifically within the South Alligator Group and lower parts of the Finniss River Group. Of particular stratigraphic importance for mineralization are the Wildman Siltstone of the mount Partridge Group and the Koolpin Formation, Gerowie Tuff, and Mount Bonnie Formation of the South Alligator Group and the Burrell Creek Formation of the Finniss River Group as well as the Tollis Formation (Figure 22) (Snowden Report, 2008). Goldfields hosted within these units include Pine Creek, Mount Todd, Howley, Golden Dyke, Maud Creek and Brocks Creek gold fields (Ahmad & Hollis, 2013). Descriptions of these prospective host units are briefly summarised as follows.

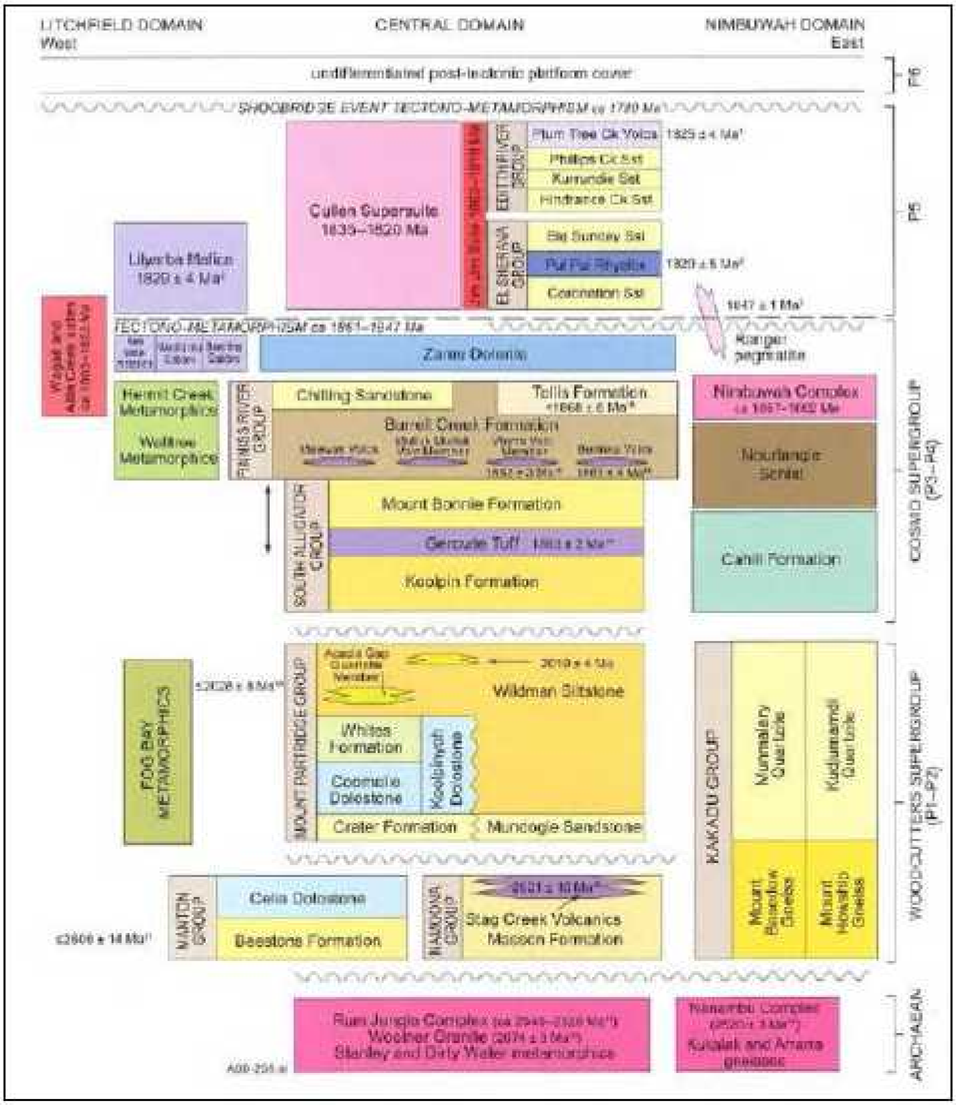


Figure 7-2: Summary Stratigraphic Chart of the Pine Creek Orogen

Source: Ahmad & Hollis, 2013

The Wildman Siltstone is the upper most unit of the Mount Partridge Group of the Woodcutters Supergroup and consists of a succession of laminated banded silty pyritic carbonaceous phyllite with minor sandstone and tuff beds, with an overall thickness of approximately 1,000 m. This unit is unconformably overlain by the Koolpin Formation of the South Alligator Group (Ahmad & Hollis, 2013).

The South Alligator Group is the oldest member of the Cosmos Supergroup and is divided into three units (Koolpin Formation, Gerowie Tuff, and Mount Bonnie Formation). Compositionally this group consists of a succession of iron rich sedimentary rocks, tuff, carbonate rocks, shale, greywacke and siltstone (Snowden Report, 2008). The Koolpin Formation is the lowermost unit of the South Alligator Group. It consists of sulphidic and carbonaceous argillite, ferruginous chert, ironstone, silicified dolomites and phyllitic mudstones which were deposited in a low energy environment (Ahmad & Hollis, 2013). The Koolpin Formation varies in thickness from less than 300 m to in excess of 1000 m. The Gerowie Tuff is up to 750 m thick and is comprised of mudstone, siliceous shale, siltstone and tuff with subordinate amounts of laminated cherts and carbonaceous siltstones. Minor quartz nodules and iron rich sedimentary sequences are additionally recognised (Ahmad & Hollis, 2013).

Numerous semi-conformable sills of pre-orogenic Zamu Dolerite intrude the Koolpin Formation and the Gerowie Tuff and vary in thickness from several meters to a few hundred meters. The Mount Bonnie Formation is the uppermost unit of the South Alligator group and consists of greywacke, carbonaceous siltstone, chert, tuff and ironstone and with a variable thickness between 150 and 400 m thick (Ahmad & Hollis, 2013).

The Burrell Creek Formation is the lowermost sequence within the Finniss River Group and is comprised of a thick (<3000 m) sequence of turbiditic sediments including greywackes, siltstones and mudstones (Snowden Report).

The Tollis Formation is the youngest member of the Finniss River Group and is host to several gold deposits including Maud Creek, Mount Todd, and Quigleys deposits (north, south, extended) (Ahmad & Hollis, 2013). This unit has limited aerial extent and consists of a succession of interbedded mudstone, slate, metagreywacke and minor felsic volcaniclastic shale that was conformably overlies the Burrell Creek Formation. This unit has previously been attributed to the El Sherana Group, however more recent interpretations of have placed this unit within the Finniss River Group (Ahmad & Hollis, 2013).

7.2 Property Geology

The Maud Creek gold field lies approximately 20km to the east of Katherine and lies within the south-eastern part of the Pine Creek Geosyncline (Figure 7-3). The Maud Creek goldfield hosts the historic Maud Creek Mine and the Maud Creek deposit (historically known as the Gold Creek deposit) (AngloGold Report, 2000). Proterozoic rock units in the Maud Creek area comprise the Tollis Formation, Maud Dolerite, Dorothy Volcanics (formerly Dorothy Volcanic Member), Edith River Volcanics, and Kombolgie Formation.

The Maud Creek deposit is hosted within the Tollis Formation which outcrops in the centre of the northwest of the Maud Creek area. This unit is typified by thin to thick beds of alternating greywacke and mudstone, minor conglomerate, altered mafic to intermediate volcanic rocks and banded ironstone. The Dorothy Volcanics consist of basaltic lava, pyroclastic rocks, tuffaceous sediments and sills and locally lie in faulted contact with the Tollis Formation (Ahmad & Hollis, 2013). To the east of the Maud Creek Deposit, the Maud Dolerite intrudes the Tollis Formation and outcrops as irregular bodies of up to 200m in width (Snowden Report, 2008).

In the northern portion of the Maud Creek area, felsic volcanics of the Edith River Group are unconformably overlain by the fluvial sediments of the Kombolgie Formation. In the south and west the Tollis Formation is masked by Cambrian Antrim Plateau Volcanics and Tindall Limestone.

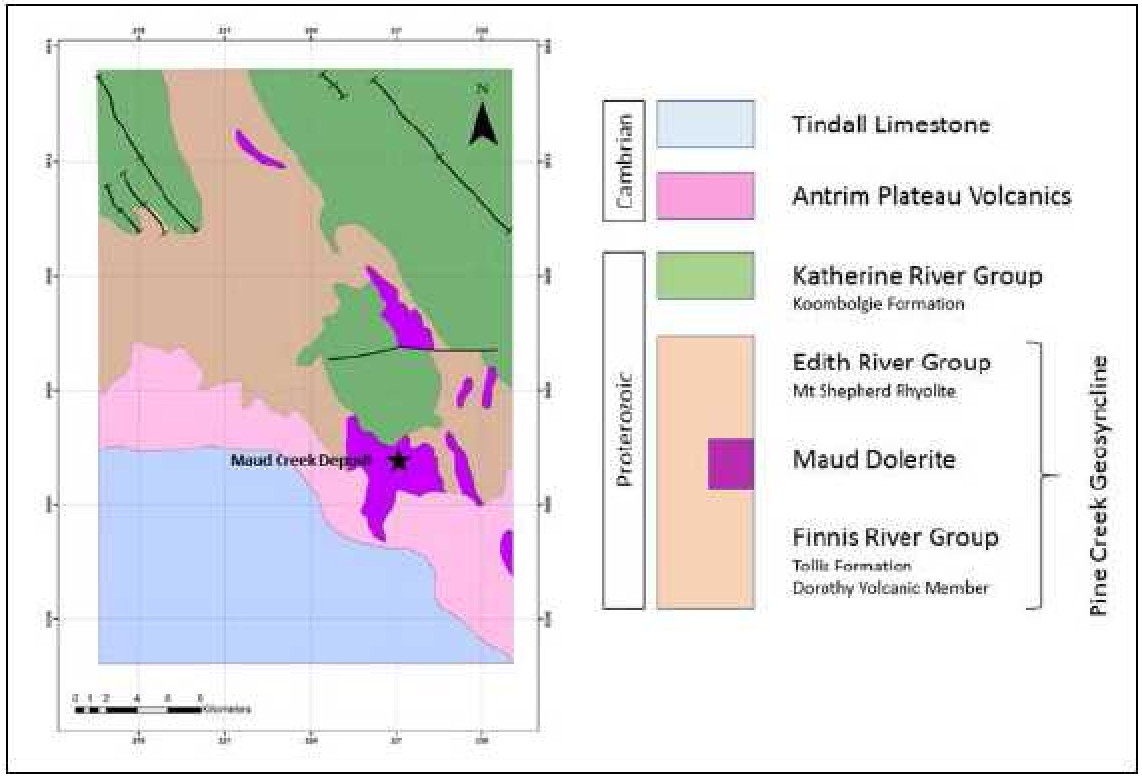


Figure 7-3: Location Map of the Maud Creek Deposit

7.2.1 Property Mineralization

The Maud Creek deposit is hosted within the Tollis Formation of the Finniss River Group. Mineralization is associated with a north-south trending Gold Creek Fault Zone (GCFZ) that forms the contact between mafic tuffs of the Dorothy Volcanics to the east and sedimentary rocks of the Tollis Formation to the west. The GCFZ and primary Maud Creek mineralization dips steeply to the east (65-75). The GCFZ is characterized by intense deformed and brecciated to catallactic zone up to 10 to 15m width (AngloGold Report, 2000). The GCFZ and Maud Creek mineralization is associated with stockworks and massive quartz veining, silica flooding and brecciation as well as intense graphitic and chloritic alteration (Ahmad & Hollis, 2013). Additional alteration recognised includes silica, carbonate, fuchsite and haematite. The contact zone deposit geometry has been defined as lenticular in shape with a steep plunge (70-80) to the south-east. This principal mineralized zone extends approximately 250 m north-south, and ranges in width from several meters to up to 50m width. The deposit remains open at depth (Snowden Report, 2008).

Mineralization is recognised to extend beyond the contact vein lodes, with dispersion up to 25m into the hanging wall tuff and 5m into the footwall sediments. Outside of the primary vein deposit minor hanging wall micro breccia zones are recognised predominantly occurring proximal to minor faulting parallel to the GCFZ (AngloGold Report, 2000).

Away from the main contact fault zone, gold is recognised within a sub-vertical shear zone which lies proximal to the contact of the Maud Dolerite (Snowden Report, 2008). Mineralization is less continuous in this zone, with the absence of any major vein lode systems evident.

Gold occurs within the deposit as both free gold and as refractory gold in pyrite and arsenopyrite (Snowden Report, 2008). Sulphides can constitute up to 5% of the deposit with pyrite and arsenopyrite and gersdorffite (NiAsS) recognised. These sulphides form as disseminations as well as massive intervals containing up to 50% pyrite (Ahmad & Hollis, 2013). Quartz makes up the remaining gangue mineral of the deposit assemblage.

Ahmad & Hollis, 2013, Pine Creek Orogen: Ahmed M and Munsen TJ (compilers). Geology and Mineral Resources of the Northern Territory, Northern Territory Geological Survey, Special Publication 5.

7.3 Deposit Mineralization

The Maud Creek deposit is hosted within the Tollis Formation of the Finniss River Group. Mineralization is associated with a north-south trending Gold Creek Fault Zone (GCFZ) that forms the contact between mafic tuffs of the Dorothy Volcanics to the east and sedimentary rocks of the Tollis Formation to the west. The GCFZ and primary Maud Creek mineralization dips steeply to the east (65-75°). The GCFZ is characterized by intense deformed and brecciated to catallactic zone up to 10 to 15m width (AngloGold Report, 2000). The GCFZ and Maud Creek mineralization is associated with stockworks and massive quartz veining, silica flooding and brecciation as well as intense graphitic and chloritic alteration (Ahmad & Hollis, 2013). Additional alteration recognised includes silica, carbonate, fuchsite and haematite. The contact zone deposit geometry has been defined as lenticular in shape with a steep plunge (70-80°) to the south-east. This principal mineralized zone extends approximately 250m north-south, and ranges in width from several meters to up to 50m width. The deposit remains open at depth (Snowden Report, 2008).

Mineralization is recognised to extend beyond the contact vein lodes, with dispersion up to 25m into the hanging wall tuff and 5m into the footwall sediments. Outside of the primary vein deposit minor hanging wall micro breccia zones are recognised predominantly occurring proximal to minor faulting parallel to the GCFZ (AngloGold Report, 2000).

Away from the main contact fault zone, gold is recognised within a sub-vertical shear zone which lies proximal to the contact of the Maud Dolerite (Snowden Report, 2008). Mineralization is less continuous in this zone, with the absence of any major vein lode systems evident.

Gold occurs within the deposit as both free gold and as refractory gold in pyrite and arsenopyrite (Snowden Report, 2008). Sulphides can constitute up to 5% of the deposit with pyrite and arsenopyrite and gersdorffite recognised. These sulphides form as disseminations as well as massive intervals containing up to 50% pyrite (Ahmad & Hollis, 2013). Quartz makes up the remaining gangue mineral of the deposit assemblage

8 Deposit Types

The contribution from gold deposits in Proterozoic sedimentary basins to total gold production has increased markedly over the past two decades, both globally and within Proterozoic basins in Australia. Consequently, many Proterozoic basins are now considered high priority exploration targets.

8.1 Deposit Models

A variety of genetic models, ranging from magmatic through hydrothermal to syngenetic, have been postulated in the past for the formation of gold deposits in the Pine Creek Geosyncline (Figure 8-1). Gold and base metal mineralization in the Pine Creek Geosyncline is commonly associated with granite intrusions and have often been classified as high temperature contact aureole deposits. A secondary host rock control has also been suggested due to the association of gold mineralization with carbonaceous metasedimentary rocks.

However, much of the gold mineralization occurred after the main intrusive event, the intrusion of the Cullen Batholith, and the relationship of gold mineralization and carbonaceous rocks is not the most important control on mineralization. More recently, authors have argued that gold mineralization is structurally controlled; occurring in brittle ductile structures at the greenschist amphibole facies boundary and hence has an epigenetic origin (Partington & McNaughton, 1997).

In places, e.g. The Cosmo Howley area, duplex thrust folds with buckle folding or basin and dome structures appear to be more significantly mineralized. The presence of shear systems linking anticlines higher in the sequence also appears to have provided the ideal fluid focusing mechanisms to localize gold bearing fluids.

Accepting that gold deposits of the Northern Territory have a structurally controlled mesothermal setting, then on the basis of host rock and mineral association they can be divided into seven types:

Gold quartz veins, lodes, sheeted veins, stockworks, saddle reefs (Pine Creek Orogen)

Gold ironstone bodies (Tennant Inlier)

Gold in iron rich sediments (Pine Creek Orogen, Tanami)

Polymetallic deposits (Iron Blow, Mt Bonnie)

Gold PGE deposits (South Alligator River area)

Uranium gold deposits (Pine Creek Orogen, Murphy Inlier)

Placer deposits

Over half of the gold occurrences are gold quartz vein deposits.

Native gold is the main mineral and is commonly present as micron sized grains; coarse nuggets are rare.

Gold is commonly associated with pyrite, arsenopyrite and pyrrhotite and in places with minor base metal sulphides. Quartz, chlorite, sericite and carbonates are the common gangue minerals in the gold quartz deposits.

All gold deposits in the Northern Territory show some structural control at the regional and deposit scales, with most deposits within the Pine Creek Orogen trending northwest southeast. Base metal veins in the Pine Creek Orogen strike significantly differently than the gold veins, suggesting different discrete mineralizing events. They are interpreted to be syngenetic.

Most deposits show a preference for competency contrast situations in dilatant or low pressure zones, such as anticlinal crests, recurrent shear zones and necking zones. Gold mineralization is invariably late, occurring after orogenic events.

Common factors for most gold deposits include:

- Gold deposits are nearly all in low grade, sub greenschist to lower greenschist facies regionally metamorphosed sediments (commonly greywacke siltstone shale)

- Anticlinal hinges and shear zones are generally the most favourable loci

- Subsequent to regional metamorphism and deformation, the metasediments were intruded by I Type granite and the gold mineralization are within the contact metamorphic aureole

- Fluid inclusion data suggest the involvement of moderate to high salinity fluids in temperature range from 200 300°C

- Stable isotope data suggest a magmatic/metamorphic origin of these fluids

Five main types of mineralization have previously been recognized within the Pine Creek Orogen. These include:

- Sheeted and stockwork quartz vein systems located along major anticlinal hinges in the Mount Bonnie and Burrell Creek Formations and to a lesser extent, the Gerowie Tuff. Mineralization is hosted by carbonaceous or sulphidic host rocks (Woolwonga) or along zones of competency contrast between greywacke and shale (Enterprise, Union Reefs, Goodall, Alligator, Faded Lily, Howley, Big Howley, Yam Creek and Fountain Head) or dolerite (Bridge Creek). Axial planar quartz veins have been identified in some deposits (Enterprise and Woolwonga) . Stratabound quartz reefs occur in most of these deposits, and may develop into saddle reefs along fold hinge zones (Enterprise, Union Reefs and Fountain Head);

- Sediment hosted stratiform gold mineralization and quartz sulphide vein hosted stratabound gold mineralization in cherty ironstone and carbonaceous mudstones of the Koolpin Formation (Tom' s Gully, Cosmo Howley, Golden Dyke and Rising Tide) or the Gerowie Tuff (Brocks Creek);

- Stratiform, massive to banded, sulphide silicate carbonate mineralization in the Mount Bonnie Formation (Mt Bonnie and Iron Blow);

- Sediment hosted stratiform and stratabound gold mineralization in cherty, dolomitic and sulphidic shales of the Mount Bonnie Formation, with sheeted quartz sulphide veins (Rustler' s Roost); and

- Sheeted or stockwork quartz feldspar sulphide veins hosted by Zamu/Maud Creek Dolerite sills (Maud Creek, Howley, Howley South, Bridge Creek and Kazi). Most gold mineralization in the Pine Creek Orogen occurs within the South Alligator Group, especially above the Middle Koolpin Formation, and in the lower parts of the Burrell Creek Formation. At Maud Creek gold mineralization is hosted by the Tollis Formation that represents the uppermost unit of the El Sherana Group and unconformably overlies the Burrell Creek Formation. Most of the fold associated deposits were probably formed during intrusion of granitoids such as the synorogenic Cullen Batholith and the Burnside Granite.

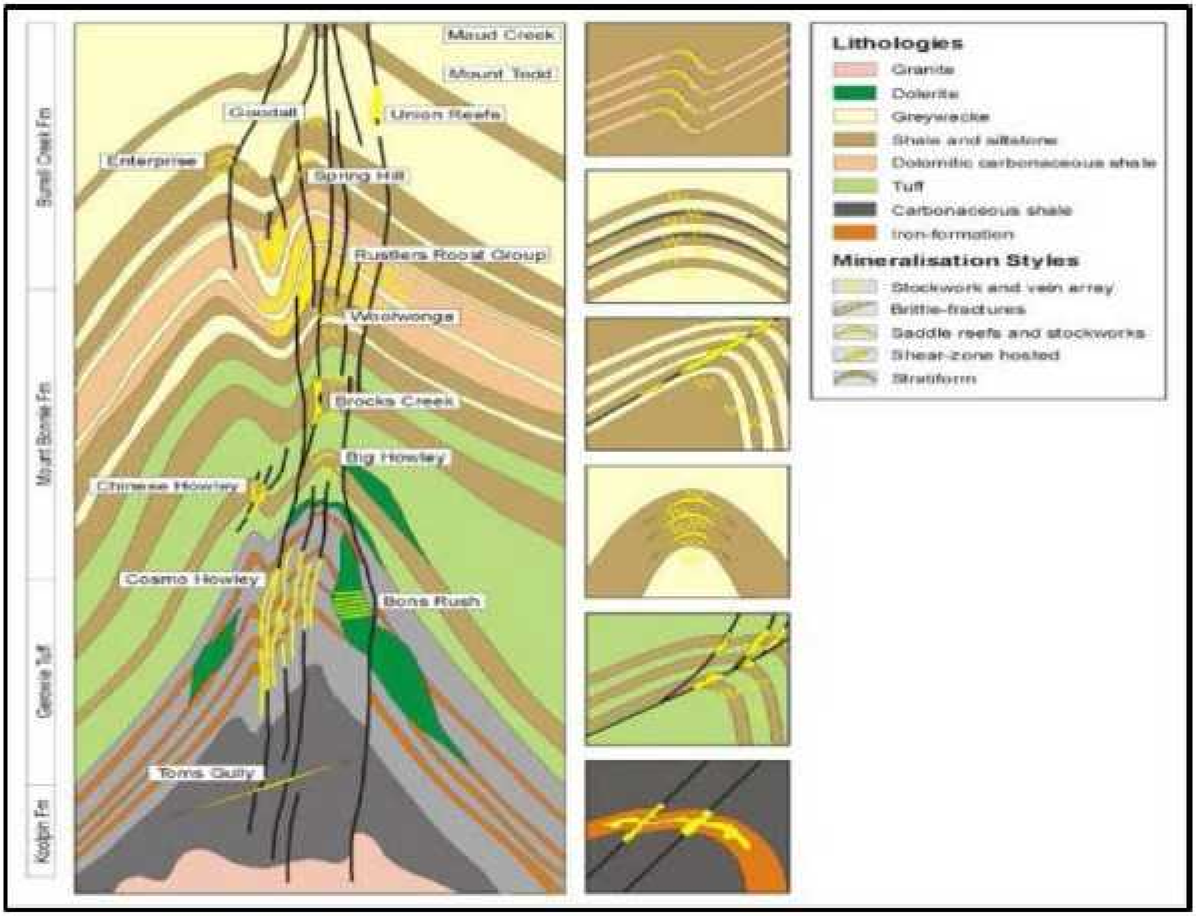
The most important regional scale exploration vectors to the orogenic style of gold mineralization are:

- The position of the biotite isograd in the contact metamorphic aureole of the Cullen granitoids. The biotite isograd needs to be mapped out carefully in areas of exploration interest and exploration focused on the biotite albite epidote contact metamorphic zone

- NNW NW oriented anticlinal axes appear to be the most productive. However, exploration cannot be totally restricted to anticlines in this orientation, as other anticlines or even synclines may be mineralized

Strongly interbedded and contrasting rock types (e.g., greywacke siltstone) particularly in the upper parts of the stratigraphy in the Mount Bonnie and Burrell Creek Formations in particular

Carbonaceous or iron rich lithologies in proximity to indications of gold mineralization. Such lithologies and any veins within them need to be mapped out carefully to help locate potential trap sites for economic gold mineralization.



Pine Creek Orogen (Sener, 2004)

Figure 8-1: Structural – stratigraphic model for Newmarket Gold deposits

8. 2 Structural Models

Assuming that the majority of gold deposits within the Pine Creek Orogen are structurally controlled and mesothermal/orogenic (cf. Groves et al. 1998) in origin, it is likely that the known gold deposits are associated with regional shear zones and fault systems that were formed during orogenesis. By analyzing maps displaying total magnetic intensity (TMI) data, a number of continuous, NNW trending first order faults can be defined within the sedimentary dominated rock sequences of the Burnside tenement area (Figure 8 2).

The majority of known gold deposits within the tenement area are spatially associated with the first order, NNW trending shear zones. It is therefore likely that these first order shear zones acted as conduits for epigenetic gold bearing fluids during/after orogenesis and they control the distribution of gold mineralization known in the tenement area. Additional factors such as the presence of the South Alligator Group, proximal antiformal hinges (e.g., Cosmo Howley) or converging secondary shear zones (e.g. Crosscourse) would also play an important role in localizing gold mineralization.

The major shear zones are separated by rock sequences that regularly preserve NNW trending, doubly-plunging anti-formal hinges with no clear evidence for strike slip deformation along these NNW trending structures. South of the Burnside granite area, a series of NE trending shear zones and faults have also been defined (Figure 8-2). Based on preserved asymmetries of rock sequences either side of these NE trending faults, dextral dominated strike slip deformation possibly occurred along these relatively later structures.

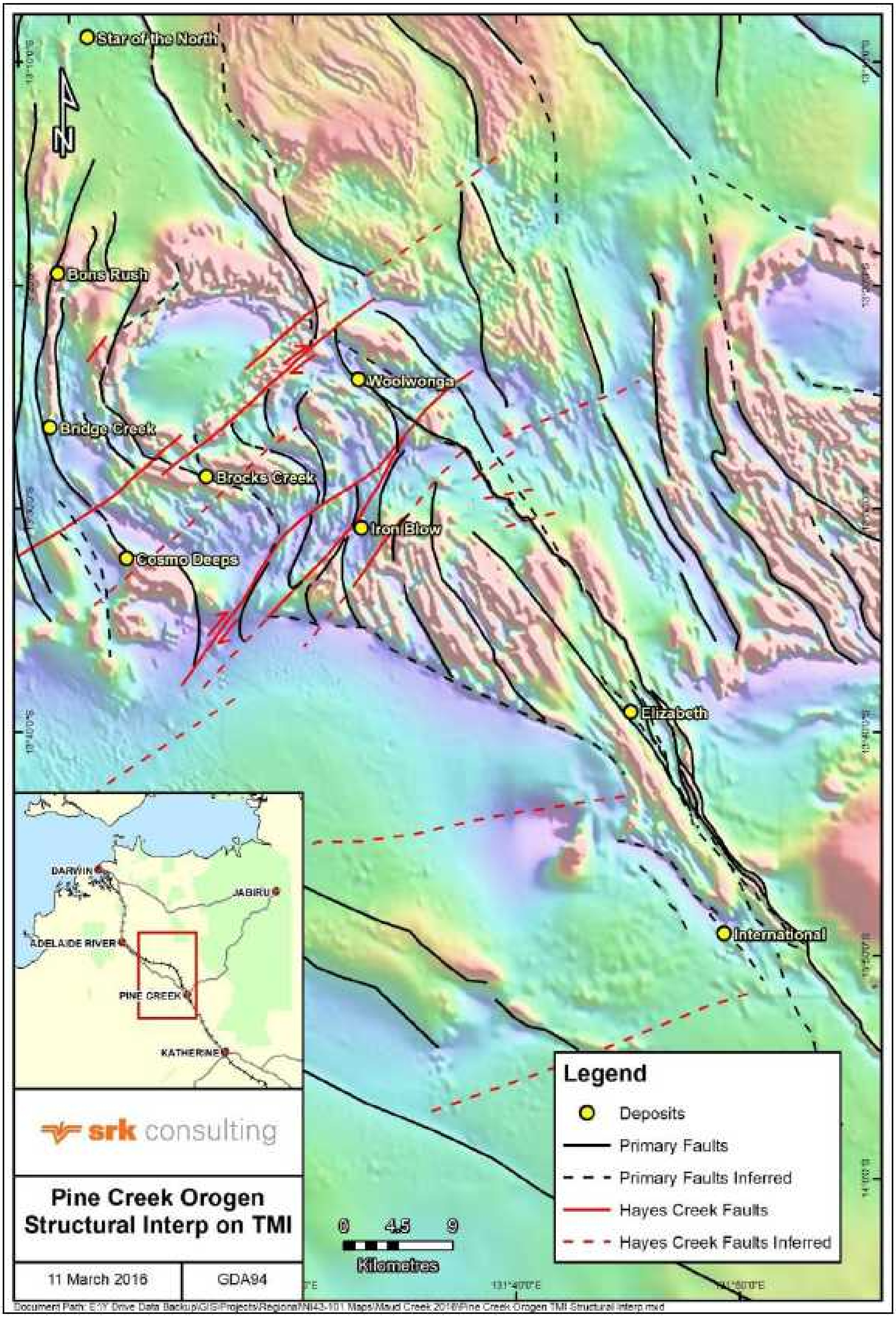


Figure 8-2: Pine Creek Regional structural interpretation

9 Exploration

9.1 VTEM Airborne Survey

A total of 590 line kilometers of VTEM survey were flown in the Maud Creek area (Figure 9-1) in 2011 covering an area of approximately 300 km^2. Line spacing was usually 200 m in the northern area (NW SE direction) and at 400 metre line spacing (NE SW direction) in the southern part of the area (Table 9-1). Southern Geoscience completed an initial interpretation and report of the survey data and Newmarket Gold geologists incorporating geology and available geochemistry reinterpreted this report (Figure 9-2).

Sixteen strong conductive targets and two moderately conductive targets were identified in the Maud Creek block. It should be noted that the strength and quality of conductors is significantly reduced from those defined at Burnside and Moline. The flat lying limestones and basalts that mask the underlying Proterozoic rocks likely play a significant role in the anomaly definition.

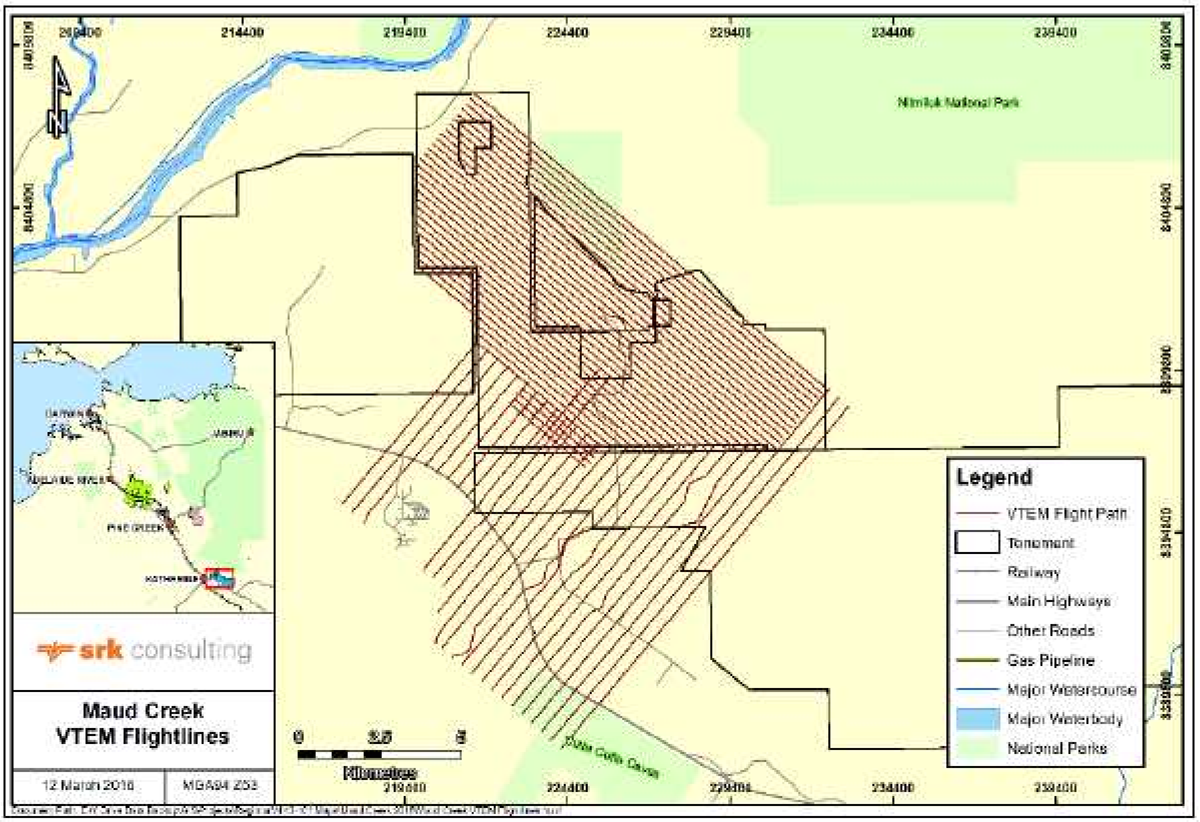


Figure 9-1: Maud Creek Property with VTEM flight lines

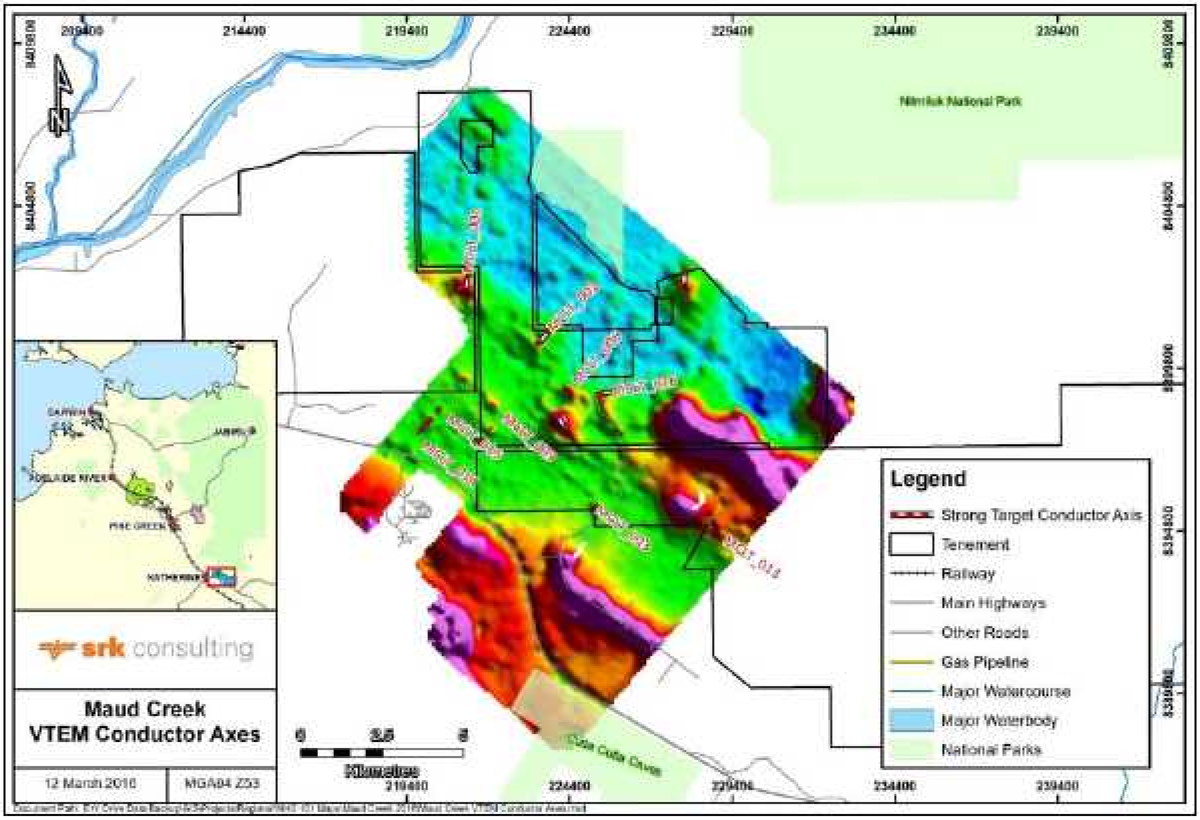


Figure 9-2: Strong VTEM Conductors on a 35 hertz conductor base map

The 25 m line spacing airborne magnetometer/ radiometric survey flown by Kalmet in 1997 clearly defines the extent of the younger Atrium basalts (Figure 9-3). Their signature produces a distinct noisy mottled effect on various manipulations of the magnetic data. The Maud Dolerites appear to have distinct magnetic anomalies along the margins of the body but the centre of the body appears to be magnetically quiet. It is quite possibly a differentiated intrusive.

Late, generally northeast trending dolerite dykes are readily apparent on the aeromagnetic image. Disruptions can be seen in these dykes indicating that later faults have created minor offsets.

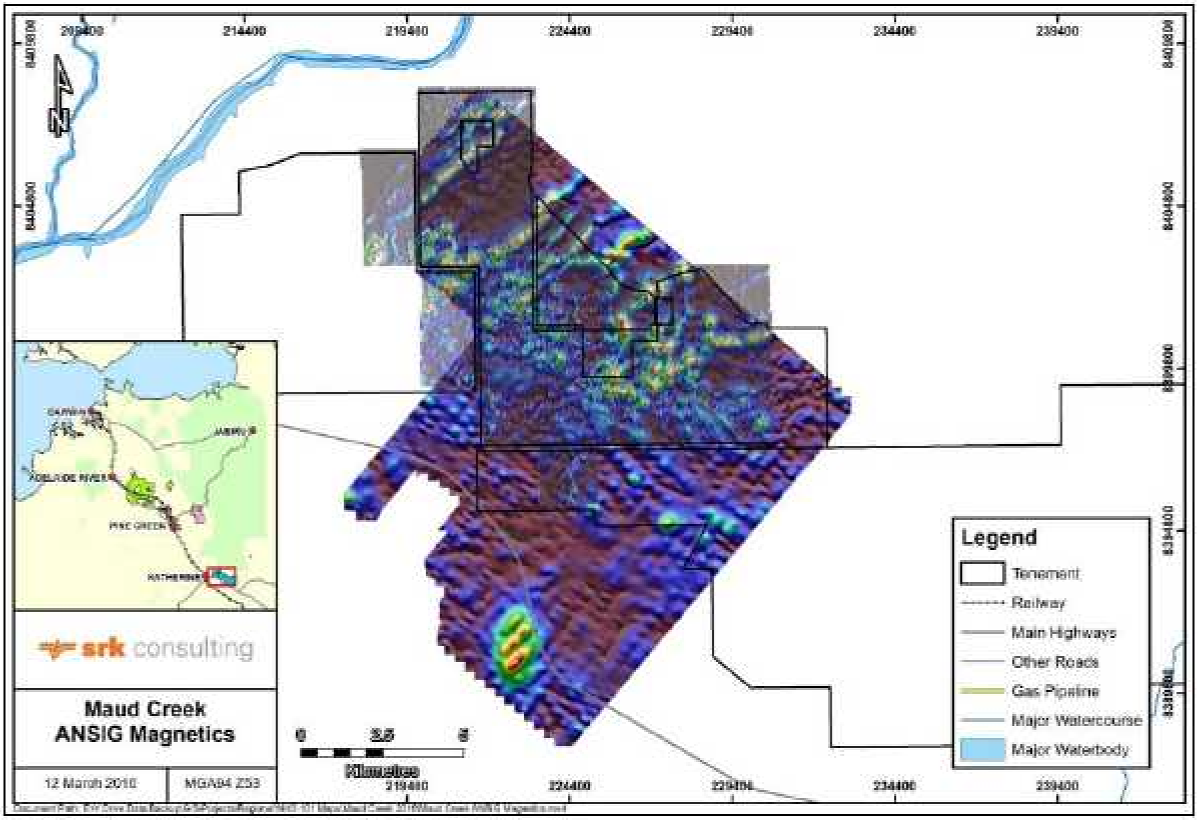


Figure 9-3: Merged 1997 and 2012 Aeromagnetic data with strong VTEM conductors (Card, 2012)

Table 9-1: VTEM conductor prioritization – Maud Creek survey area

Conductor #	SG Priority	Priority	Length (m)	Surface Work	Host Fm	Comments
MCLT005	1	1	400-600	No	lmstn	SW dipping, near a lmstn o/c, quartz veining noted in the area, previously Maxwell modeled, don' t have the model. Low ground between o/c. Deep seated. NW strike, no distinct magnetic signature. Weak Pb and Au in soil anomalies. Geochemistry likely masked by lmstn. May be on a NNW trending lineament
MCLT003	2	2	200	No	lmstn	Embayment in lmstn? Flanking linear magnetic anomaly to the north. Dyke? Right on the edge of High res mag survey area. Soil color anomaly on Google? Possibly on a north south lineament. Isolated conductor
MCLT007	3	2	200	No	lmstn	Just north of a stream, soil color anomaly, weak magnetic anomaly indicated on high resolution magnetic. Distinct direct magnetic anomaly. Work this first. Deep seated. Need to model
MCLT008	3	3	200	No	lmstn	Small magnetic anomaly, near o/c
MCLT009	3	3	200	No	lmstn	Small magnetic anomaly, larger anomaly to the NW, covered by high resolution magnetic. Near o/c to the north. Deep seated >200 m?
MCLT010	3	3	200	No	lmstn	Small magnetic response, on the exploration tenement. o/c to the north soil color anomaly to the north
MCLT011	3	3	200	No	lmstn	Small magnetic response, on the exploration tenement. o/c area
MCLT012	3	3	200	No	lmstn	Small coincident magnetic anomaly, on exploration tenement. o/c in area
MCLT014	3	3	200	No	lmstn	Small coincident magnetic anomaly, on exploration tenement. North of o/c
MCLT006	3	3	200	No	lmstn	Close to a NE trending lineament located to the SE. SE dip. Pb in soil anomaly
MCLT016	3	3	600-800	No	lmstn	SG says probably culture. On road but migrates to the south. South dip. Fence line but no different from other fences. Walked the road. Lots of road metal contamination from mine area. Limestone cover. o/c to south. Magnetic anomaly coincident on eastern line. Noisy data.

Conductor #	SG Priority	Priority	Length (m)	Surface Work	Host Fm	Comments
MCLT015	4	3	200	No	lmstn	Very similar to 007, 008, 009, 010, 011, 012, 014. Determine the cause of 007 first. Very small target. Non magnetic, o/c area. Edge of soil sample survey
MCLT002	4	3	800-1000	No	lmstn	Possibly thick conductive overburden. On a north south lineament. No distinct magnetic signature. NE strike. On the NW trending base metal trend but striking the wrong direction
MCLT001	4	2	800-1000	No	lmstn	Possibly conductive overburden. No distinct magnetic signature. On exploration tenement. Changes strike direction from ENE to NS. On base metal anomaly but strikes the wrong way. Low ground. Possible o/c area. Possible north south lineament with Chessmen and Red Queen to the north
MCLT004	4	1	200	No	lmstn	At the south end of a linear magnetic feature that hosts Maud Creek to the north. Also on a NW trending magnetic feature that may have an association with the base metal in soil anomaly. Watch out for buffalo. Increased vegetation in the area. Slight soil color anomaly
MCLT013	4	2	200	No	?	Very small and narrow, very isolated response. Weak magnetic anomaly. Possible east west lineament. Outside soil survey area.
MCLT017	4	1	600-800	Yes	Maud dolerite	At old Maud Creek workings. Change in strike direction to conductor. ENE to NNE. Associated with magnetic anomaly. Northern response outside property boundary . Highly conductive regolith. Alluvials in area
MCMT002	NP	1	200	No	lmstn	SE of a large magnetic anomaly interpreted to be in a regional NE trending structure. Appears to be in a nature park, karst caves. Interesting target but deep seated and too many social issues. Ignore

9.2 Stream sediment survey

In the mid 1990' s a fairly extensive stream sediment survey (approximately 173 samples) was carried out in the Maud Creek area. Samples were taken along various drainage systems at sample spacing of 500 m or closer. It is believed that a BLEG analysis was carried out with elements such as silver, copper, lead, zinc, bismuth, molybdenum and antimony also determined (Table 9-2).

In 2012 it was decided to stream sediment sample the 600 km2 area of EL 28902 in order to quickly determine its mineral potential. Arnhem Exploration Pty Ltd were contracted to carry out the survey.

A total of 164 stream sediment samples were collected along streams at approximately 1 km spacing. Figure 9 4 displays the sample distribution over the tenement area. Note that there is some duplication/overlap in the areas sampled with the 1996 97 survey. Samples were collected from several sites at any particular collection point in order to alleviate point anomaly sources. Each site was photographed. Samples were sieved to 75 microns in the field. A total of 9 duplicate samples were taken for QAQC purposes.

A visual examination of the QAQC sample duplicates indicates that the gold values exhibit good repeatability while the other elements returned acceptable levels of repeatability. It should be noted that one sample exhibited marginal repeatability in lead and copper.

A correlation matrix of the 2012 survey data population indicates that there are weak correlations between gold and arsenic (0.38), bismuth (0.36), nickel (0.42) and tin (0.37) . There are significantly higher correlations with silver and copper (0.87), lead (0.88), antimony (0.92), arsenic (0.61) and barium (0.66) . It is suspected that several mineral deposit types are present in the area and that the correlation relationships are reflecting those differing styles.

A brief statistical look at the results for selected elements from the 1996 97 survey and the 2012 survey in Table 9-2 reveals the following:

Table 9-2: Comparative statistics between 1997 and 2012 soil survey results

Year	Mean Ag Grade (ppm)	Mean As Grade (ppm)	Mean Gold grade (ppb)	Mean Cu Grade (ppm)	Mean Pb Grade (ppm)
1997	0.021	4.1	0.699	33.1	55.2
2012	0.026	2.0	0.695	19.6	26.4

There is a good agreement between gold and silver values for both surveys. There is a wide divergence for copper, lead and arsenic. The base metal soil anomaly located south of the Maud Creek deposit skews the 1997 survey results. If both surveys were to be merged then the latter 3 elements would have to be normalized before they could be plotted and displayed effectively.

Underlying geology plays a distinct role when it comes to interpreting the stream sediment data. Largely Cenozoic and Mesozoic materials that overlay Proterozoic lithologies and/or Cambrian age sediments or volcanics underlie the significant land area to the east of the Maud Creek deposit. The masking effect of these younger sediments has likely diluted any anomalous effects from underlying Proterozoic rocks so subtle anomalies need to be field checked. It is conceivable that there are windows through the younger sediments that expose the Proterozoic as they do at the Copper Breccia occurrence that are located at the east end of the original Maud Creek tenements.

The gold results generated a number of anomalies. Most obvious is the cluster of anomalous samples in the Maud Creek deposit area and that area, which is underlain by Maud Dolerite. The 1997 survey did not produce an extensive gold anomaly in this area. It is suspected that sampling the finer sediments in 2012 produced more consistent results.

The 1997 survey produced a strong and extensive Au anomaly in the Red Queen/Chessmen area (northwest part of the property). This area was not re sampled in 2012 but the area to the southwest was and the gold anomaly appears to extend into this area.

In the area that the geology map indicates (Figure 9-4) there is extensive cover a number of gold anomalies occur. These all have anomalous multi element associations and all need to be ground checked.

Anomaly 1 is a cluster of two samples that are anomalous in gold, iron, chromium, arsenic, lead, molybdenum, tin and uranium. A preliminary interpretation of regional aeromagnetic data indicates that this anomaly may be situated within the southern limits of the Maud Dolerite unit (as defined by magnetics). SRK' s structural interpretation of the region carried out for Kilkenny Gold in 1998 indicates several large ENE trending structures that cut through this target area.

Anomaly 2 is one sample that is highly anomalous in gold (9.7 ppb) as well as molybdenum, bismuth and arsenic. The area should be resampled with either 3 or 4 stream sediments or a small soil grid established to cover the upstream area. The anomaly area does not exhibit any distinct magnetic signature

Anomaly 3 is a cluster of 4 samples that is anomalous in gold, copper, bismuth, arsenic, lead, chromium, uranium, tin, silver and barium. It is interpreted to be in the southern extension of Maud Dolerite. SRK' s structural interpretation of the region carried out for Kilkenny Gold in 1998 indicates several large ENE trending structures that cut through this target area. Anomaly 4 is located at the very northeast part of the survey area and is anomalous in gold, silver, iron, bismuth, arsenic, lead, chromium, uranium, tin and silver. The area is likely underlain by Tollis Fm. The regional magnetic data indicates it is proximal to a northeast trending dolerite dyke. Northwest trending structures can be interpreted. The gold in stream sediment anomaly appears to occur at the intersection of two major structures according to the SRK interpretation.

Anomalies 5, 6, 7, 8, 9 and 10 are one and two point anomalies. They all need to be ground checked. The extensive base metal anomaly defined from the Kalmet 1997 stream sediment and soil surveys is clearly defined on the normalized copper and lead stream sediment map. It would appear that is may extend over a distance of 30 35 km. It obviously weakens to the east and west but this may be a function of increased thicknesses of younger cover. The base metal anomalous area is also defined with elevated values in barium, cadmium, chromium, iron, potassium, magnesium, manganese, nickel, tin and zinc.

The base metal anomaly in stream sediments and soils doesn' t appear to be directly related to any individual VTEM anomaly and there does not appear to be a correlation to any particular magnetic response. The area of the soil anomaly is largely underlain by young Cambrian aged volcanics, although one area is underlain by Maud Dolerite. If the geochemical response comes from the Proterozoic rocks then there is some sort of mechanism allowing it to percolate through the younger volcanics. Looking at the airborne magnetic and radiometric data it would appear that the base metal anomaly occurs at or very close to the younger volcanic – limestone contact. The geochemical anomaly can be traced for >10 km in a generally east west direction.

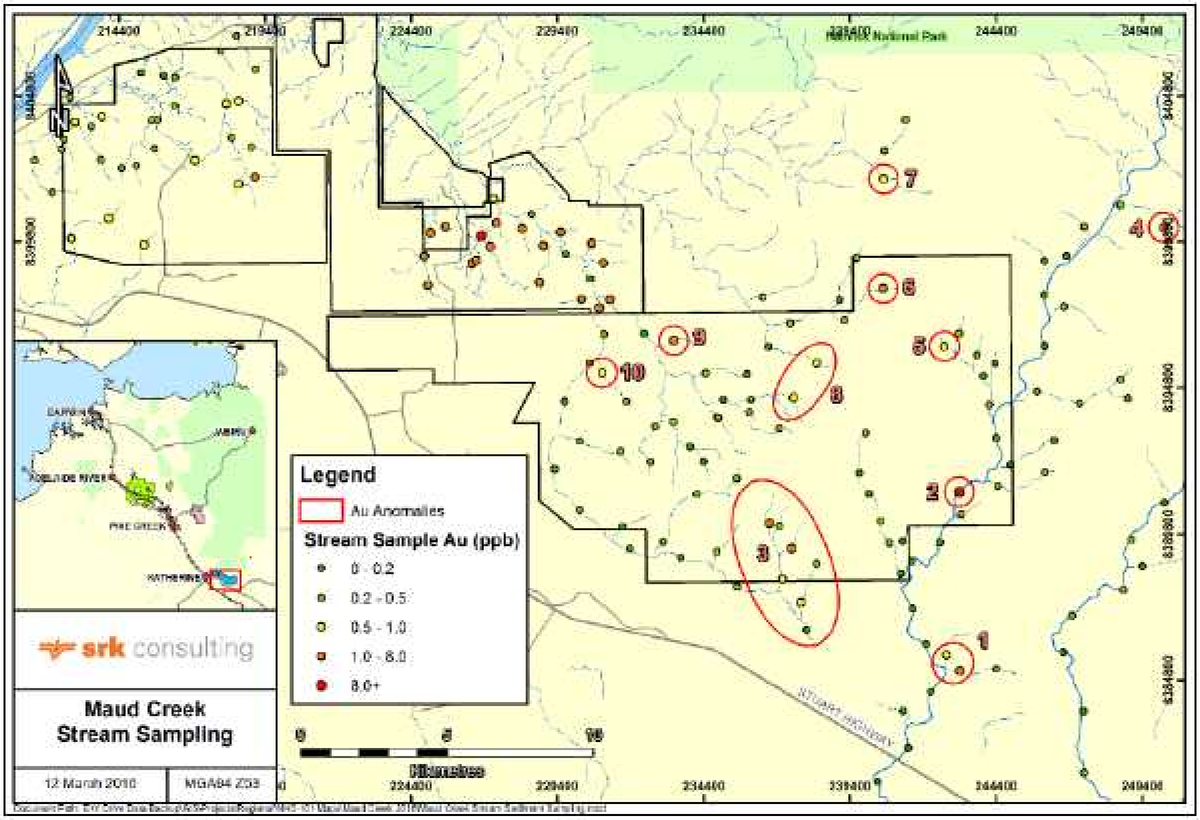


Figure 9-4: Maud Creek Area – 2012 stream sediment survey – selected gold anomalous areas

9.3 Soil sampling surveys

In 1997 Kalmet carried out an extensive soil survey over the Maud Creek area on lines 400 m apart and samples taken at 25 metre intervals but composited to 50 metre intervals (Figure 9-5). Samples were analyzed for gold, silver, copper, lead, zinc and antimony. A number of anomalous areas were defined but the coarse line and sample spacing didn' t allow for accurate directional interpretations.

Early in 2012 fourteen target areas were identified and given a letter designation. These did not include the VTEM anomaly reprioritization. The objective was to determine through rock chips, soil sampling and mapping the location of possible future exploration drillholes.

Prioritized Targets:

Anomaly A UTM 221500E 8404400N

- Two gold in soil anomalies. The eastern one is about 1200 m long with coincident antimony while the western one is 400 m long with coincident antimony over 800 m. Both have elevated arsenic values. It is interpreted that these are not overbank stream sediment related. No significant base metal anomalies.
- No significant VTEM anomaly associated with the geochemical anomalies
- The magnetic data indicates a possible NE trending dyke (late dolerite)
- Possible K anomaly. Distinct uranium/thorium anomaly,
- Possible north south structure interpreted from Total Count and SRK structural data.
- Chessmen and Red Queen occurrences to the north, possibly on the same north south structure.
- Likely underlain by Antrim volcanics, basalt

- Nothing distinct on Google image.
- May be the source area for the distinct and widespread gold in stream sediment anomaly in the area.

Anomaly B UTM 221500E 8406400N

- Gold in soil anomaly over 1200 m running north south with coincident antimony and arsenic over a 400 m length. There are streams to the north and south so there is a possibility that there may be contamination from overbank stream sediments.
- SRK structural geology indicates Tollis limestones and a north south structure, which may connect up with Target A.
- Very weak VTEM channel 25 response. Possible NNW trending feature.
- Magnetics indicate an interpreted NE trending dolerite dyke. Possible north south folds.
- Small but distinct uranium/thorium anomaly.
- Nothing distinct on Google image
- Need a compilation of past work at Red Queen and Chessmen prospects.

Anomaly C UTM 221000E 8407400N

- The south part of the anomalous area has a stream close by so there may be overbank stream sediment contamination. One anomalous gold in soil sample to the west. There are elevated arsenic and antimony results to the north. Possible overbank contamination.
- No distinct VTEM response.
- Possibly the NW end of a NNW trending structure.
- Magnetics indicates a NE trending late dolerite dyke.
- There is a north south trending structure to the east.
- Distinct Total Count (TC) anomaly with some potassium contributing.
- Nothing distinct on Google image. Obvious stream

Anomaly D UTM 222200E 8400000N

- Two anomalous Au in soil results. Soils here were not analyzed for arsenic, antimony or base metals.
- VTEM indicates a possible north south structure on channel 25
- Likely underlain by Tollis limestone No distinct magnetic features
- Radiometrics indicates that the target is on the south edge of a major NW trending TC anomaly
- SRK map indicates possible volcanics at the edge of limestones.

- UTM 224000E 8401000N

- Gold in soil anomaly over 400 800 m running north south. Elevated arsenic and antimony. On the edge of base metal anomaly which is part of the major NW trending response.
- Kangaroo Flats prospect to the north. Don' t know anything about this prospect.

- Distinct magnetic anomaly, unknown cause.
- No distinct radiometric anomalies.
- SRK map indicates east west structures
- Underlain by Antrim volcanics
- Nothing distinct on Google image

Anomaly F UTM 224300E 8402400N

- Gold anomaly in soils 400 800 m north south. No significant arsenic, antimony or base metals. Possibly an overbank stream sediment anomaly.
- Kangaroo Flats prospect is 500 m to the west No VTEM response
- Radiometrics: possible NW structure and possible north south structure. No distinct anomalies.
- Underlain by Antrim volcanics
- Low priority target

Anomaly G UTM 225000E 8401000N

- Two point gold in soil anomaly. No arsenic, antimony or base metals. May be overbank stream sediment.
- Near Curlies area to the SE
- Underlain by Antrim volcanics
- Distinct potassium anomaly
- SRK map indicates an east west structure
- No distinct magnetic features

Anomaly H UTM 225400E 8401600N

- Maud Creek deposit area anomaly
- Gold in soil anomaly extends 1600 2000 m north south
- Coincident arsenic and antimony with weak molybdenum and silver. No elevated base metals. Maybe contaminated from overbank stream sediments.
- VTEM weak channel 25 response trending north south
- Sediment tuff contact Antrim volcanics to the immediate west and south
- Radiometrics indicates a distinct north south low as well as NE and NW lineaments.
- Magnetic low feature. West of Maud Dolerite.

Anomaly I UTM 226000E 8402000N

- Gold in soil anomaly 600 m east west x 500 m north south. Coincident arsenic and antimony anomalies. Quite possibly overbank sediments.
- Chlorite Hills to the NE.
- Surprise area between targets H and I.

- Underlain by Maud Dolerite
- North of NE trending late dolerite dyke
- Weak potassium anomaly – probably Maud Dolerite signature
- Close to Maud Creek and old workings

Anomaly J UTM 226700E 8401600N

- Gold in soil anomaly 400 800 m north south trending. Coincident arsenic anomaly
- Antimony coincident over 1200 1600 m Jibaroo prospect.
- Maud Creek to the north. Possible overbank stream sediment contamination
- No VTEM response K/Th anomaly possibly caused by dolerite. NE structure indicated.
- North south structure to the east.
- Good target area. Maud look a like target – geophysically
- East west structure.

Anomaly K UTM 227600E 8401600N

- Gold in soil low order anomaly with coincident anomalous antimony and strong arsenic. Fairly widespread so may be overbank contamination. Near a stream
- North – south structure to the east
- No distinct radiometric anomalies
- No distinct VTEM anomalies
- Magnetic low, magnetic destruction?
- Underlain by Maud Dolerite

Anomaly L UTM 228200E 8401600N

- Runways prospect
- No streams in the immediate area.
- Gold in soil anomaly over 400 800 m with coincident arsenic and antimony.
- Base metals are very low order.
- Magnetic anomaly caused by Maud Dolerite
- Potassium/thorium anomaly interpreted to be caused by Maud Dolerite
- Bracketed by north south structures.
- Good structural target. East margin of Maud Dolerite

Anomaly M UTM 226400E 8399600N

- Small isolated gold in soil anomaly with coincident arsenic and weak antimony. Right on several streams so quite possibly overbank contamination.
- Drovers prospect, anomaly at south end of prospect

- Distinct north south structure, good target area
- Intersecting NE and NW trending structures
- Potassium/thorium anomaly caused by Maud Dolerite.
- Linear magnetic low trending NNE

Anomaly N UTM 227500E 8399600N

- Single point anomaly with coincident antimony and arsenic anomaly to the west.
- No proximal streams
- No distinct base metal anomalies
- Potassium/thorium interpreted north south structure
- Maud Dolerite overlain by Antrim volcanics

- Possible NE trending magnetic anomaly interpreted to be caused by a dolerite dyke.
- No distinct VTEM anomaly

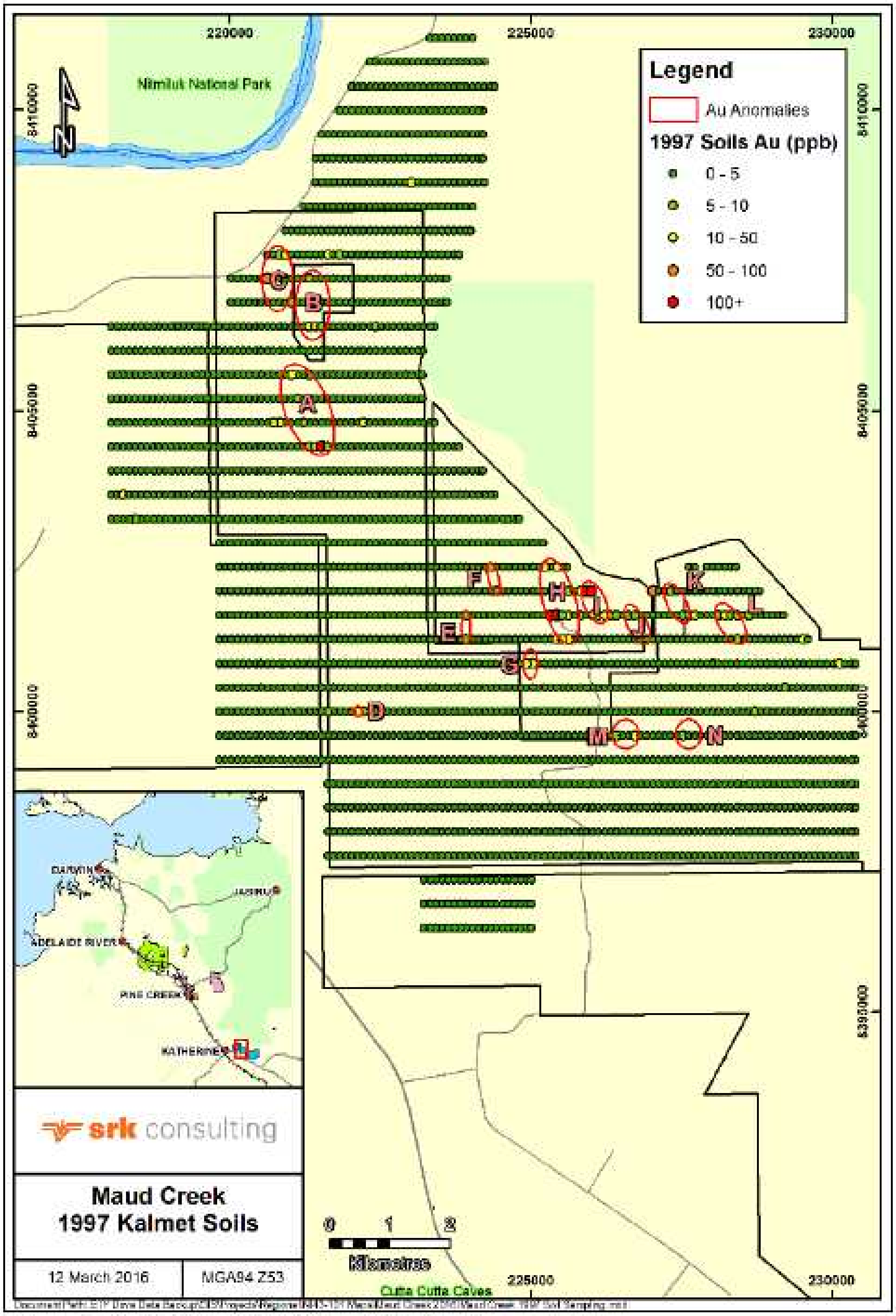


Figure 9-5: Location of soil anomalies requiring further work

A limited field program carried out later in 2012, designed to investigate the 14 target areas, defined the following:

Anomalies A and B

Anomalies A and B were originally contoured with a north south bias, however, field examination indicated that the anomalous gold in soil geochemical values should be contoured in a north northeast direction (Figure 9-6). When this was done two narrow gold in soil geochemical trends with coincident arsenic and antimony anomalous values were defined, coincident with two 020 025° trending felsic dykes and/or silicification / brecciation.

Additional detailed soil sampling was recommended and ultimately carried out in 2012.

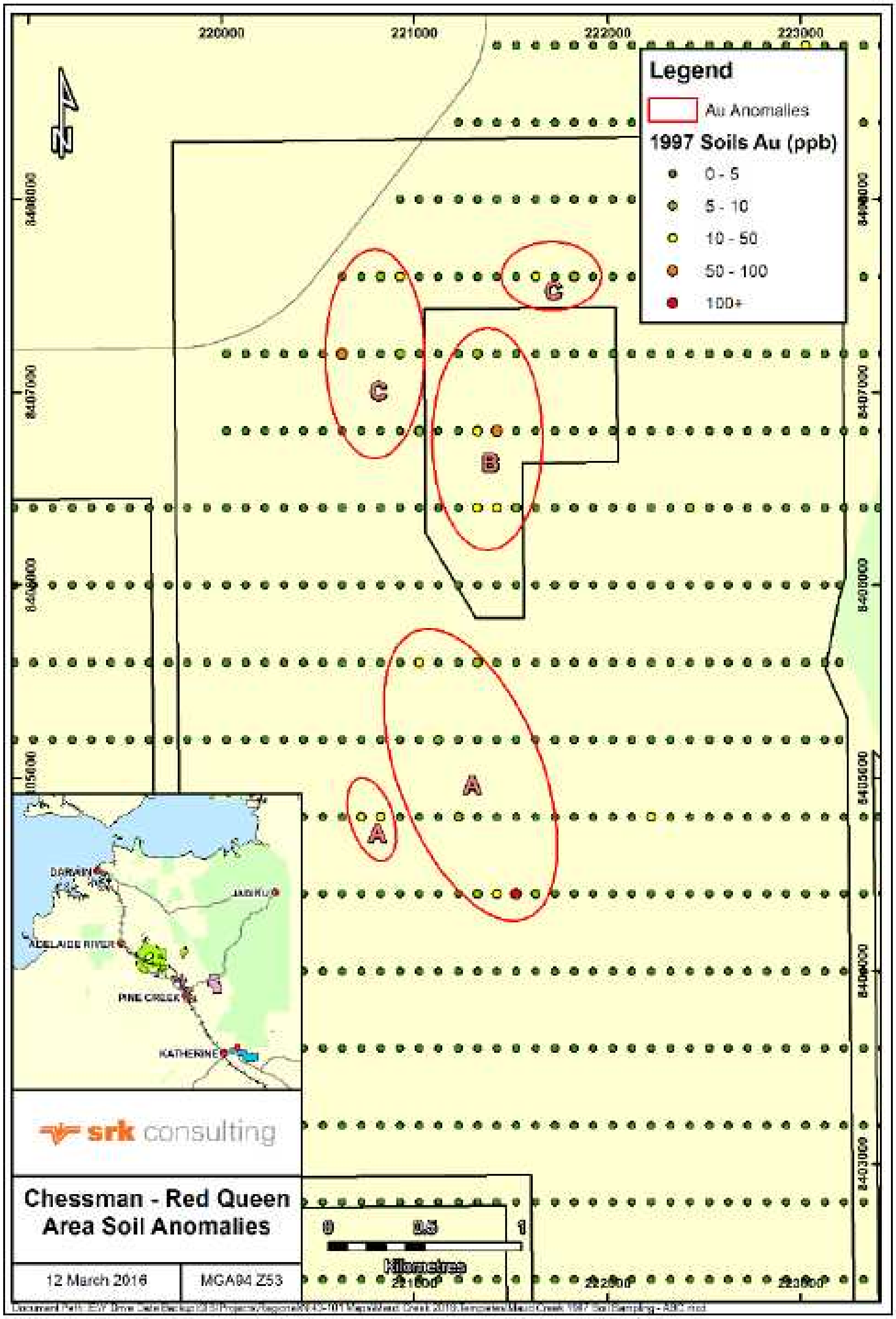


Figure 9-6: Red Queen – Chessman Area – Soil anomaly designation

The Chessman and Red Queen occurrences both occur on the northernmost trend (B) (Figure 9-6). The Chessman occurrence, located at UTM 8406600N; 221460E, is located south of a 58.6 ppb gold soil anomaly, and the Red Queen occurrence, located at UTM 8406230N; 221330E, south of a 11.3 ppb gold soil anomaly (both anomalies defined by past operators work). This siliceous zone on which both the Chessman and Red Queen occurrences are located continues southward for at least an additional 2 km, siliceous brecciated material located at anomalous gold in soil sample sites which returned 33.9 ppb and adjacent to soil sample site which returned 11.6 ppb.

Anomaly H

Anomaly H was only briefly visited as it defines the known mineralization at the Main Maud Creek deposit (Figure 9-8). The strongest gold in soil analytical results occur exactly on what is now the Maud Creek open pit mined in 2000. The northern and southern extensions of the gold in soil anomaly defined the mineralized trend of the Maud Creek deposit.

Chlorite Hills (Northern part of Anomaly I)

The strong gold in soil anomalous results at the north end of Anomaly I are probably defining the gold mineralization associated with the Chlorite Hills veining (Figure 9-7). Two fences of drillhole collars were located along a 25 x 25 metre grid spacing. Not all of these drillhole collars are in the Newmarket Gold database, however, those that are were drilled at a 270° azimuth, parallel to the observed quartz veining. Additional drilling completed by Kilkenny Gold was oriented in multiple directions, 320°, 140°, 180°, 90° and 270° azimuths. Anomalous gold, ranging from 0.54 g/t over 4 m to 2.25 g/t over 4 m, was intersected. The assembly of the drilling database for the Maud Creek Property area is still in progress.

There were no surface indications of a north south structure as indicated by Hill 50 NL in an internal memo by Bob Watchorn, however, the drilling completed by Kilkenny with 090° and 270° azimuths did intersect anomalous gold (up to 2.25 g/t over 4 m), possibly indicating the existence of north south mineralized structures in the area.

A traverse was conducted between the Maud Creek open pit to the Chlorite Hills pits to try to determine if the mineralized quartz veins at Chlorite Hills extended to the Maud Creek deposit, however, there were no outcrop exposures until the Maud Dolerite. The contact between the Tollis Formation and the Maud Dolerite occurs at a north south oriented creek approximately 400 m east of the Maud Creek deposit. A second traverse was completed from the Chlorite Hills 040° trending pits ending exactly at the one drillhole completed at the Surprise area, possibly indicating a 040° structure. (RC hole MCT 013 completed by Newmarket Gold 2011, 270° Azimuth, 60° dip. No significant gold analytical results were returned. The entire hole is in dolerite). There was no evidence noted to support previous interpreted north south structures in this area.

O' Shea' s to Anomaly J

The O' Shea' s occurrence is located at UTM 8401920N, 0226920E. An outcrop of very fine grained siliceous rock disseminated with very fine sulphides with brecciated angular clasts to 10 cm occurs at the occurrence within coarse grained diorite. The silicified zone trends 30° with several pits excavated along its trend. Quartz vein mapping by Kilkenny Gold define NE trending veins in this area with additional minor NW trending veins.

The main workings strike north south with drilling completed east west. A fair amount of historical drilling has been completed at the occurrence. RC drilling in 1998 targeted the NE trending shearing and associated workings. The mineralization intersected a 2 to 4 metre wide quartz/sulphide shear within the dolerite dipping 60° to 70° to the southeast.

A traverse at 30° was completed directly to Anomaly J (Jilleroo occurrence) gold in soil geochemical target arriving onto a series of shallow pits (Figure 9-7). These pits were investigating siliceous hematitic, highly altered rock and quartz veining and/or brecciation in hematitic diorite. Minor malachite was noted along fractures. Shearing and quartz veining were striking 135°. Two sub parallel 135° trending shears were observed.

The Jilleroo occurrence may be occurring at the intersection of the continuation southward of the 30° O' Shea' s shear and the 135° structures.

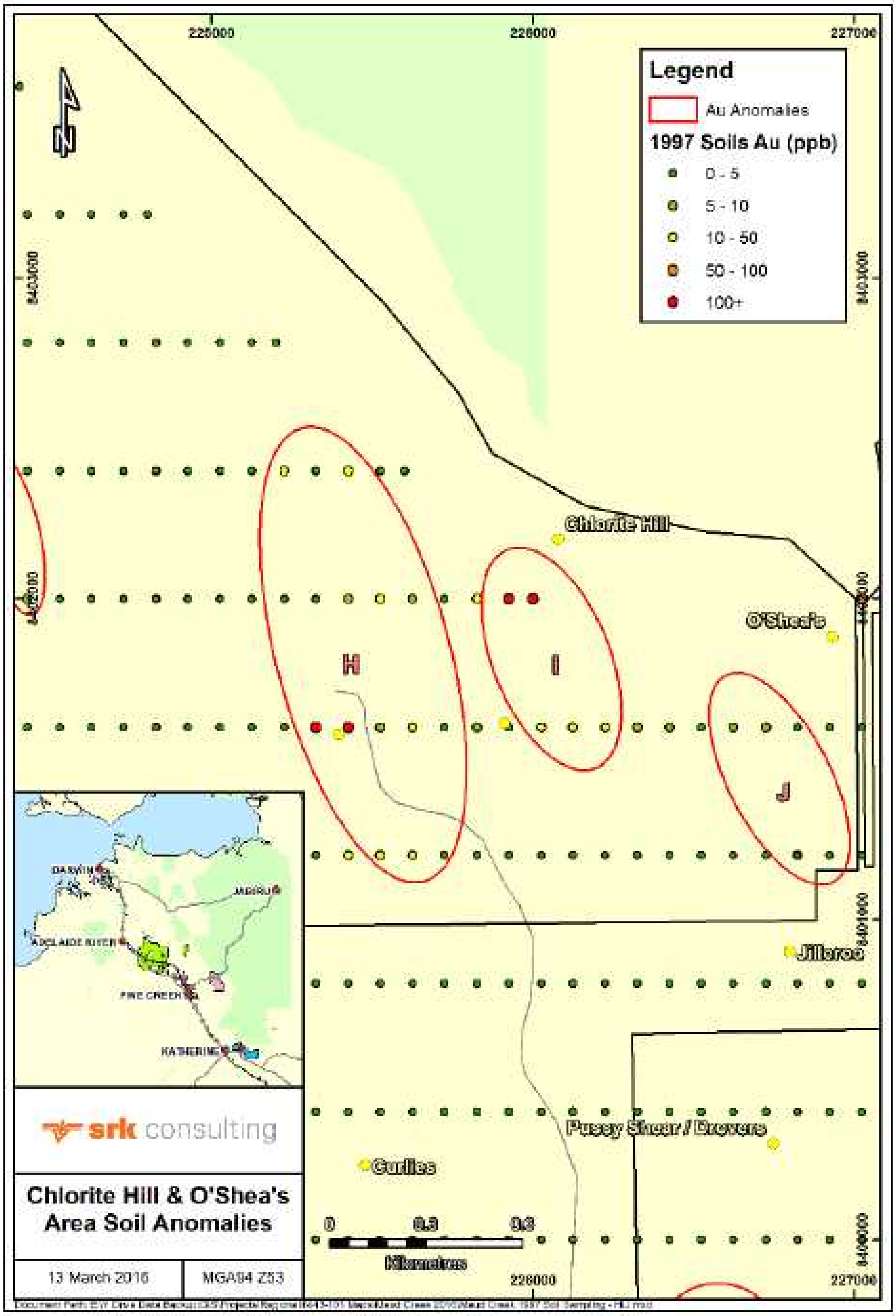


Figure 9-7: Chlorite Hills and O' Shea' s soil anomaly designations

Anomaly G, south end of H and south end of I to O' Shea' s single point gold in soil

Drawing a straight line lining up the eastern most anomalous gold in soil analytical result at Anomaly G (11 ppb) to the south end of Anomaly H (Maud Creek) (16 ppb) to Anomaly I (Surprise) (16.6 ppb) defines a 60° trending gold in soil anomaly (Figure 9-9). Extending this trend to the northeast the trend will pass immediately north of O' Shea' s and arrive at the Carpentania Pass target. The northeastern portion of this trend parallels a siliceous felsic dyke mapped in the area. This 60° trend has been noted numerous times throughout the property. The gold in soil analytical results may define a 60° crosscutting structure in the area or perhaps a fault that cuts the Maud Creek deposit at the southern end.

There has been extensive drilling completed (both RC and RAB) over the northern and southern extensions of the Maud Creek deposit by past operators. This drilling covered Anomaly G and Anomaly H. Very little drilling has been completed over the area east of The Maud Creek Deposit.

Pussy Shear Zone

A brief visit was conducted at the southern end of the Pussy shear zone area (Figure 9-8). Extremely hematitic gossanous material with a stockwork of cm wide quartz veinlets and sub crops of agglomerate, similar to that which occurs at the Maud Creek deposit were noted. A postulated north south shear zone was not confirmed. Additional prospecting should be completed along this trend to confirm Hill 50 NL' s Internal Memorandum defining a north south structure. Several narrow 135° trending shears within diorite were noted. Several hand drawn sketches (completed by Kilkenny Gold) indicate en echelon NW SE trending structures within a north-south corridor. Grab samples collected by Kilkenny returned values between <0.01 to 0.18 g/t Au.

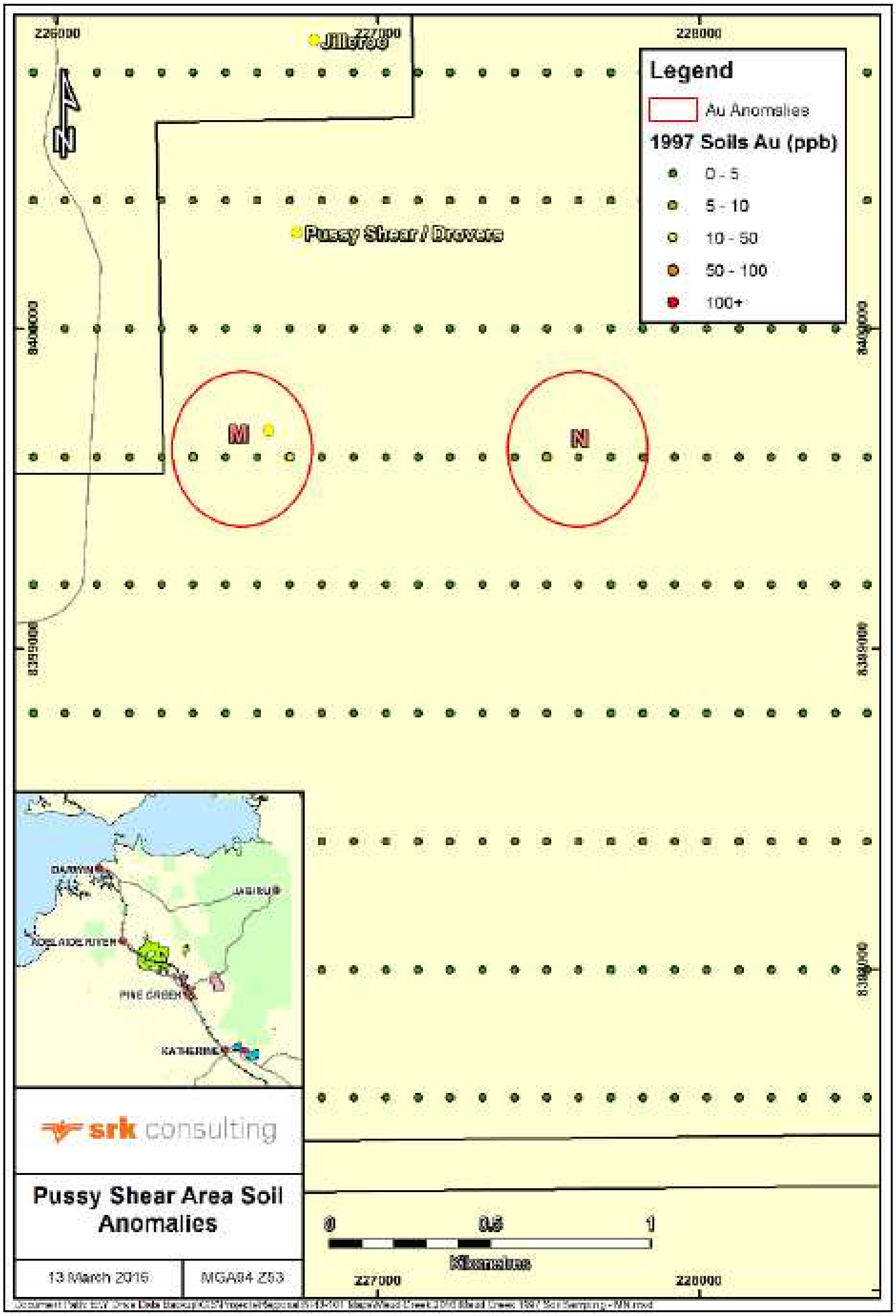


Figure 9-8: Pussy Shear zone soil anomalies

Kitten (Drivers) Anomaly

A brief visit was conducted at the Kittens (Drovers) Anomaly M area (Figure 9-9). A corridor of extensive quartz and quartz breccia float trending 60° to 70° was located. Within this corridor several discontinuous siliceous sub crop ridges, massive to brecciated trending 80° were observed. In 1998 Kilkenny Gold drilled a fence of RC holes across the southern portion of this area (17 holes totalling 842 m). Drilling did not intersect any significant gold values. Two different intrusives were intersected by the drilling (best gold value was 0.8 g/t over one metre interval along one of the intrusive contacts).

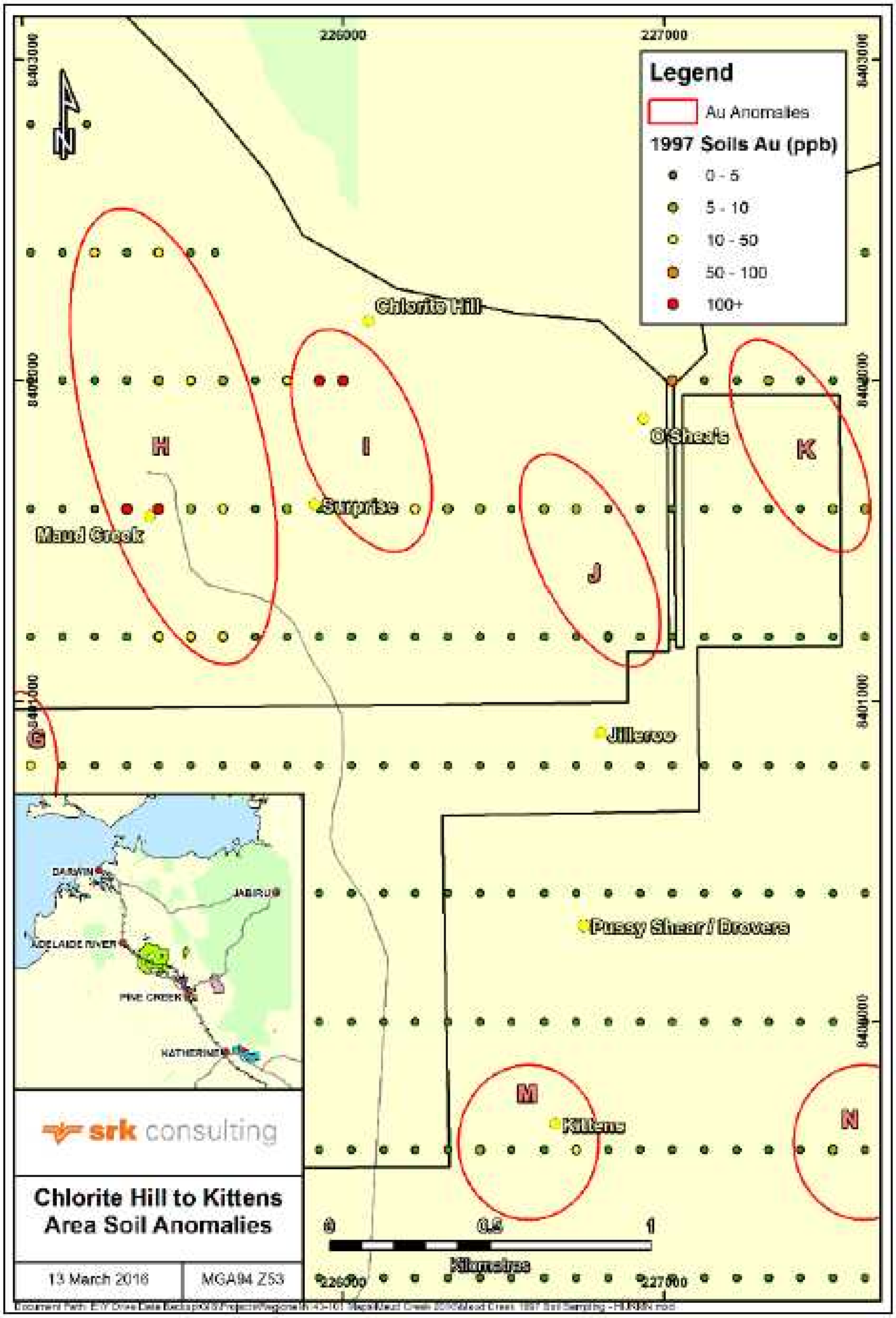


Figure 9-9: **Chlorite Hills and Kittens (Droves) soil anomaly areas**

Anomaly L

Gold in soil Anomaly L occurs at the Runways target. The area is flat non descriptive terrain with numerous float boulders of very fine grained dolerite within an area of reddish soil with patchy manganese coatings. There were no obvious indications as to the cause or source of the elevated gold in soil analytical results. Past operators completed two east west lines of drillholes across this area, which extended across Anomaly K. A total of 44 RC holes (2,534 m) were drilled. The drilling returned no significant gold results (best value 0.14 g/t Au over 4 m). Buried intrusive and tuff were intersected. The NW trending magnetic anomaly in the area appears to correspond with these intrusives.

Anomaly E

Anomaly E (defined by two 7.6 ppb gold north south correlated gold in soil analytical results) located on strike with VTEM anomaly MCLT_002 was prospected as part of the property examination. The area is flat, overlain with numerous boulders and rubble of quartzite, fine grained sandstone and chert. The overburden cover, as seen in creek beds is at least one metre thick. There was no obvious cause for the elevated gold in soil analytical results.

VTEM Anomaly MCLT_002

The area of VTEM conductor MCLT_002 is extremely flat. Minor outcrops of quartzite overlain by a thin cover of limestone were found. There was no obvious cause for the VTEM anomaly.

Subsequently, it was decided to soil sample two areas in detail, Chlorite Hills O' Shea' s and the Red Queen/Chessman area. Samples were taken along lines 100 m apart, oriented to cross cut regional structures and geology. Samples were taken 25 m apart. The ionic leach method of analysis was selected so samples were taken at shallow depths of 10 20 cm and were sieved in the field to 75 microns. ALS Chemex' s Au + ME MS41 0.0001 0.1 Au by Aqua Regia with ICP MS finish multi element package was the analytical method used.

Chlorite Hills Area

In the Chlorite Hills/O' Shea' s area a total of 596 soil samples were collected by Arnhem Exploration with assistance from Newmarket Gold field assistants. This includes 20 sample duplicates taken to monitor QAQC of the commercial lab. These were subsequently shipped to ALS Chemex facility in Darwin NT. Instructions were to have the samples analyzed using Chemex' s ionic leach ME MS23 multi element package. A visual inspection of the 20 QAQC duplicate samples indicates that the repeatability for the sample population is good with no obvious errors.

A correlation matrix of the entire population indicates strong correlation (0.77) between gold and copper and a weak to moderate correlation between gold and silver and arsenic (0.35, 0.32) . Table 9-3 displays a comparison of the mean for a few elements from the two sample areas at Maud Creek and the one VTEM target area east of Bon' s Rush in the Burnside area.

Table 9-3: Comparative soil sample statistics for Maud Creek and Bons Rush Area

Area	Mean Ag Grade (ppb)	Mean As Grade (ppb)	Mean Gold grade (ppb)	Mean Cu Grade (ppb)
Chessmen	2.67	6.5	1.13	1,270
Chlorite Hills	8.9	4.97	5.35	2,249
VTEM Anomaly Burnside Area	3.41	13.43	0.25	975

The gold values in the Chlorite Hills area are significantly higher than those from the other two areas. The same applies for silver and copper (Figure 9-10). One explanation may be that the Chlorite

Hills/O' Shea' s area has seen historic mining with some significant ground disturbance and this contamination may be partially responsible for the elevated values in multiple elements.

Nevertheless, the gold in soil values exhibit an interpreted east west trend in gold that is not obviously repeated in other elements. The O' Shea' s and Chlorite Hills areas stand out as being quite anomalous in gold and somewhat in copper. O' Shea' s is anomalous in silver and antimony.

The west side of the grid displays anomalies in silver, calcium, iron, magnesium, nickel and anomalously low in zirconium, uranium and thorium. It is suspected that this area represents a contact with Maud Dolerite, which is interpreted from magnetic data to be a differentiated intrusive unit.

The eastern margin of the grid displays anomalism in calcium, magnesium and nickel. It is suspected that this exhibits a contact with a phase of the Maud Dolerite unit.

An anomalous area at the southeast end of the grid centred on 226900E 8401400N displays elevated values in a number of elements including antimony, thorium, titanium, uranium, zirconium, lithium, iron, arsenic, copper and weakly in Au. The area needs to be ground checked.

It is suspected that multi element anomalism in the region of 226300E 8402200E may be due to overbank sediments associated with Maud Creek, which is situated immediately to the northeast. This needs to be ground checked.

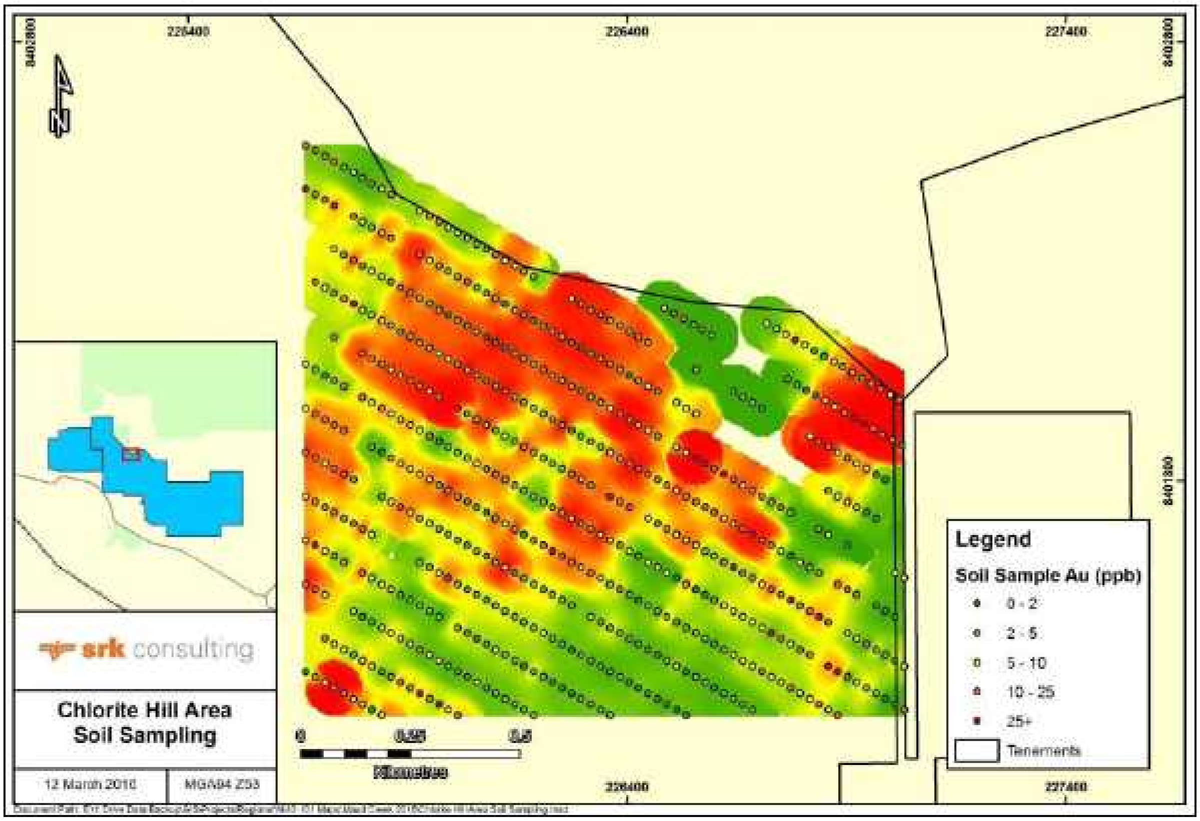


Figure 9-10: Au (ppb) for ionic leach soil results at Chlorite Hills, Maud Creek

Chessman Area

In the Chessman/Red Queen area Arnhem Exploration collected a total of 1,942 samples with assistance from Newmarket Gold field assistants (Figure 9-11). This includes 61 sample duplicates taken to monitor QAQC of the commercial lab.

A visual inspection of the 61 QAQC duplicate samples indicates that the repeatability for the sample population is good for most elements with acceptable variability between duplicate pairs. For gold values four of the duplicate pairs exhibit a variability that is outside acceptable limits. The other elements return acceptable values for the most part.

A correlation matrix of the entire population indicates a fairly strong match of 0.67 between gold and silver. The gold in soil values presents a unique distribution that is not matched with any other element. In Figure 9-11 a NNE trending anomaly extends for 2.5 km along the west side of the grid that would appear to extend beyond the grid to the NNW. A preliminary interpretation would indicate that the gold anomaly sub-parallels the west contact of a lithological unit that appears to be associated with a NNE trending syncline (graben?). In all likelihood its emplacement is structurally controlled. There is no distinct magnetic correlation with the gold anomaly although it would appear to cut through a strongly magnetic NE trending dolerite dyke located at the NW end of the grid.

Another Au in soil anomaly at the north end of the grid needs to be investigated on the ground. A highly folded magnetic anomaly located at the south end of the grid can be traced for many kilometers to the southeast. Malachite staining was noted in the area. The magnetic anomaly is coincident with contorted anomalies in copper, silver, magnesium, barium, calcium and lead and possibly a weak gold response. It is markedly negative in molybdenum, barium, tin, antimony, uranium and iron. Conceivably, this anomalous situation correlates with other significant base metal anomalies defined at the southeast end of the Maud Creek property. Further field investigation is required.

A variety of other elements are obviously defining differing lithologies that define the synclinal structure. An example is iron that defines the east and west margins of the syncline. It, along with a number of other elements also defined some folded structures within the core of the syncline. Field investigation is required to link these anomalous situations to specific rock types. The centre of the syncline has its own distinct geochemical signature and the ionic leach data along with magnetic and radiometric information can be used with great effect to help with geological and structural interpretations.

Calcium and magnesium data can be used to interpret crosscutting (NW SE) trending dykes. The VTEM data has defined a NW trending feature that correlates directly with a fence. A stronger NW trending conductive feature at the north end of the soil grid is likely associated with a dyke like feature.

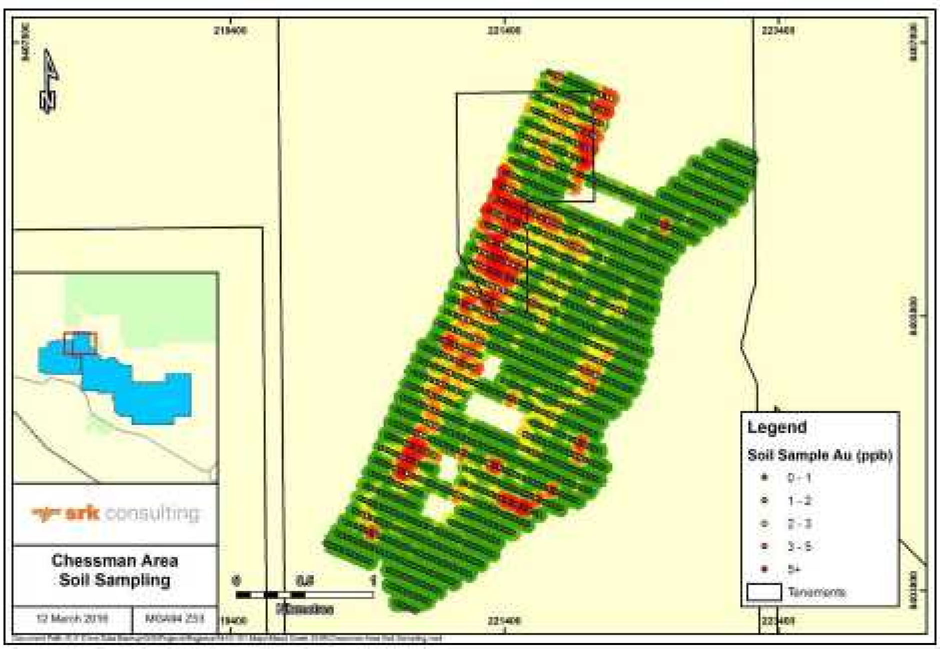


Figure 9-11: Ionic leach Au (ppb) in soil results, Chessman Area, Maud Creek

9.4 Rock Chip Sampling

During the reconnaissance prospecting of various soil anomalies a series of 58 rock samples were collected in the Red Queen/ Chessman and Chlorite Hills areas. Figure 9-12 displays the gold results.

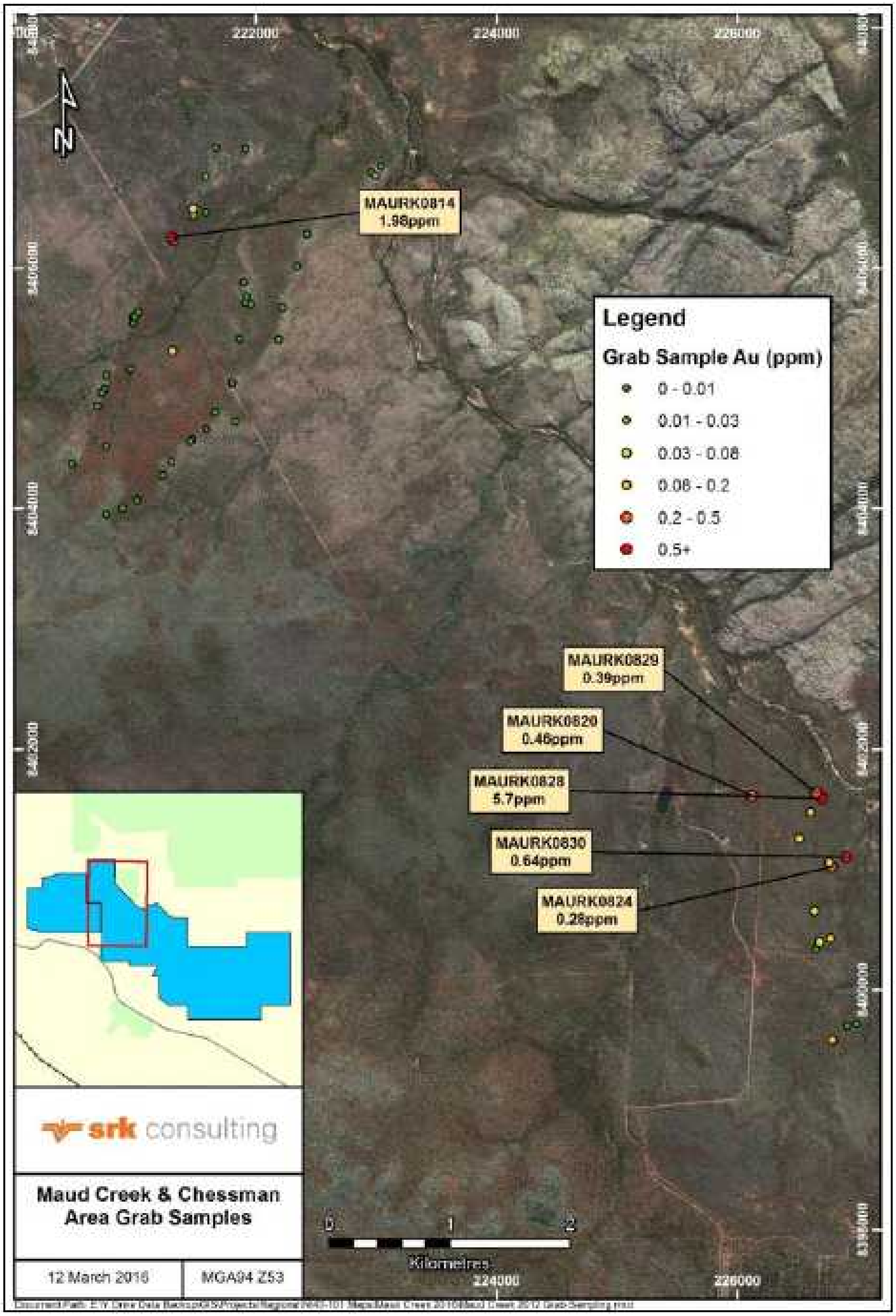


Figure 9-12: Maud Creek area gold in 2012 rock chip sample results

Anomalous gold values also display elevated values in silver, arsenic, bismuth, copper, iridium, lead, antimony, tin, tungsten and zinc. There is a distinct phosphorus depletion associated with the anomalous gold values. In the Red Queen/ Chessman area anomalous gold in rock chip values are located at the north-west margin of the syncline where the soil results are anomalous. At the Chlorite Hills/O' Shea' s area anomalous gold in rock chip samples seem to be associated with the eastern margin of the Maud Dolerite.

10 Drilling

Over 94,000 m of RAB, RC and Diamond drilling has been completed at the Maud Creek Project and surrounding areas (Table 10-1 and Table 10-2). While the drilling prior to 2011 was not completed by Newmarket Gold, a significant amount of historical data is available for review and reporting. Based on the quality of the data available, SRK see no indication that the drilling information cannot be used for Mineral Resource estimation.

Table 10-1: Drill statistics for the Maud Creek deposit

	Diamond			RC	
Company	**Period**	**# of holes**	**Meters**	**# of holes**	**Meters**
CGAO	2011	6	3,180.33	14	702
Terra Gold	2007	1	211.50	0	0
Kalmet/Hill 50/Terra	1995-97, 2001-2002, 2005	70	19,800.12	0	0
Gold					
Kalmet	1994	0	0	356	42,390.8
Kalmet	1993	0	0	36	2,212
Placer	1990-91	20	2,992.37	0	0
Placer	1989-91	0	0	36	3,503.8
Total		97	26,184.32	442	48,808.60

Table 10-2: Historical drilling by previous tenement holders

Prefix	# of holes	Type	Company	Period	Notes
DRC	44	RC	KGNL		
GRC	17	RC	KGNL		
KR	282	RAB	KALMET		
MC01 - MC25	25	DR	MCML	Dec 1993	
MCE001 - 021	21	RC	KGNL		
MCP037 - 301	265	RC	KALMET	Sep 1994 - Jun 1996	
MCP302 - 416	115	RC			
MCP417 - 488	40	RC	HILL 50	Aug 2001 - Aug 2005	
MCW	54	RC	KGNL	Apr 1998 - Nov 1998	
MD001 - 033	33	DD	KALMET	1995	
MD034 - 052	21	DD	KGNL	1997	Includes 2 wedge holes by HILL 50 (2001).
MD053 - 062	14	DD	HILL 50	Aug 2001 - Aug 2002	
MD063 - MD065	2	DD	TGML	Jul 2005 - Aug 2005	
MRB	216	RAB	KALMET	Aug 1997 - Sep 1997	
MRC	36	RC	KALMET	Jan 1993	

RAB	588	RAB	MCML	Oct 1993	
SRRC	14	RC	KGNL	May 1998	
SWM	9	RC	KALMET	Jun 1998 - Sep 1998	
TMCD	1	DD	TGML		
WD	20	DD	PLACER	Jun 1990 - Jun 1991	Includes diamond tails
					to WP holes
WP	40	RC	PLACER	Jun 1989 - Jun 1990	

10.1 2011 Drilling program

Newmarket Gold, then trading as Crocodile Gold conducted infill drilling at Maud Creek in 2011:

Diamond - Some holes were collared using PQ (122.6 mm) to allow for better recovery and to prevent collar collapse, this was then reduced to HQ once ground conditions allowed (Table 10-3).

Diamond drilling was used at Maud Creek by Newmarket Gold (Figure 10-1) to ensure accurate logging of structures, lithology, alteration and mineralization, as well as capture of geotechnical data. Several diamond drillholes were also used for metallurgical sampling.

Table 10-3: Parameters used for infill diamond drilling at Maud Creek

Deposit	Diamond			RC			First drilled	Last drilled
	Holes	Meters	Size	Holes	Meters	Size		
Maud Creek	6	3,179	HQ-PQ	14	700	5”	19 Sept 2011	16 Nov 2011

Reverse Circulation - RC drilling was used in areas where diamond drilling was not required or appropriate, for example, to test the potential mineralization to the south of the Maud Creek pit where limited drilling was identified.

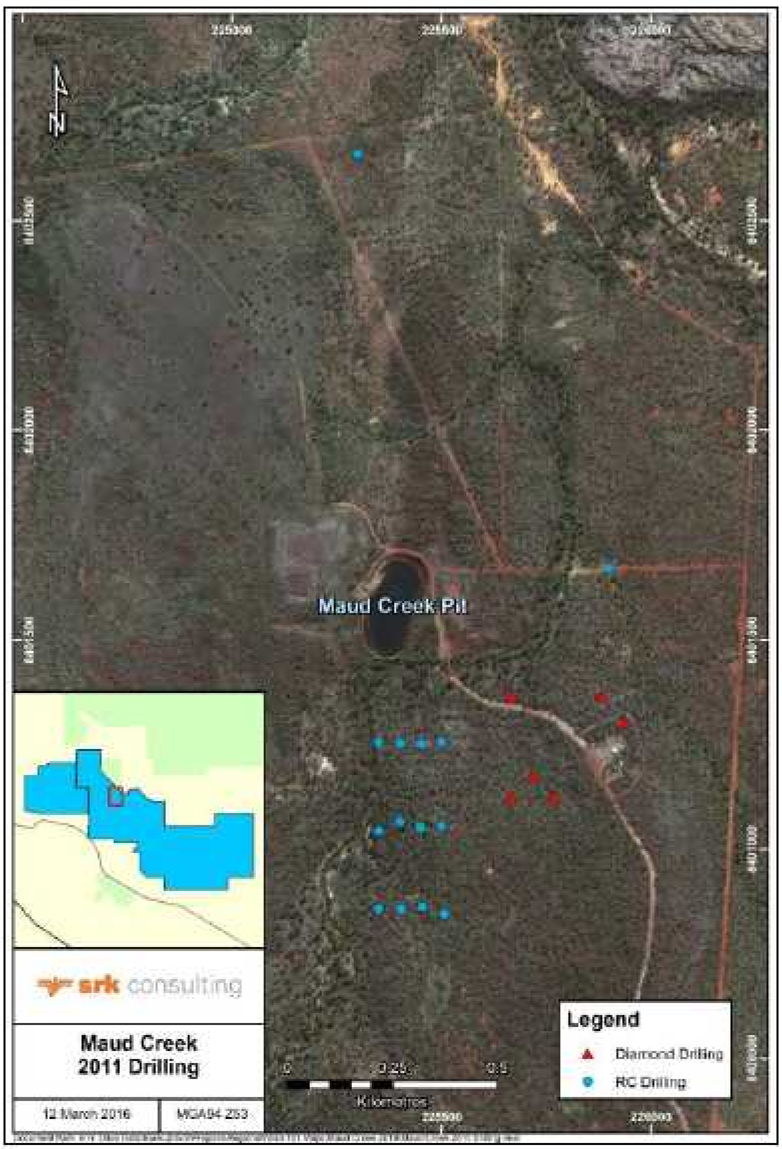


Figure 10-1: Plan view of the RC and DDH Drilling completed in 2011

The objective of the drilling program was to test the down plunge component of the Maud Creek deposit at depth. Four holes were used to test for deposit extension to the south, while two were used for verification of the resource model (Figure 10-2 and Table 10-4).

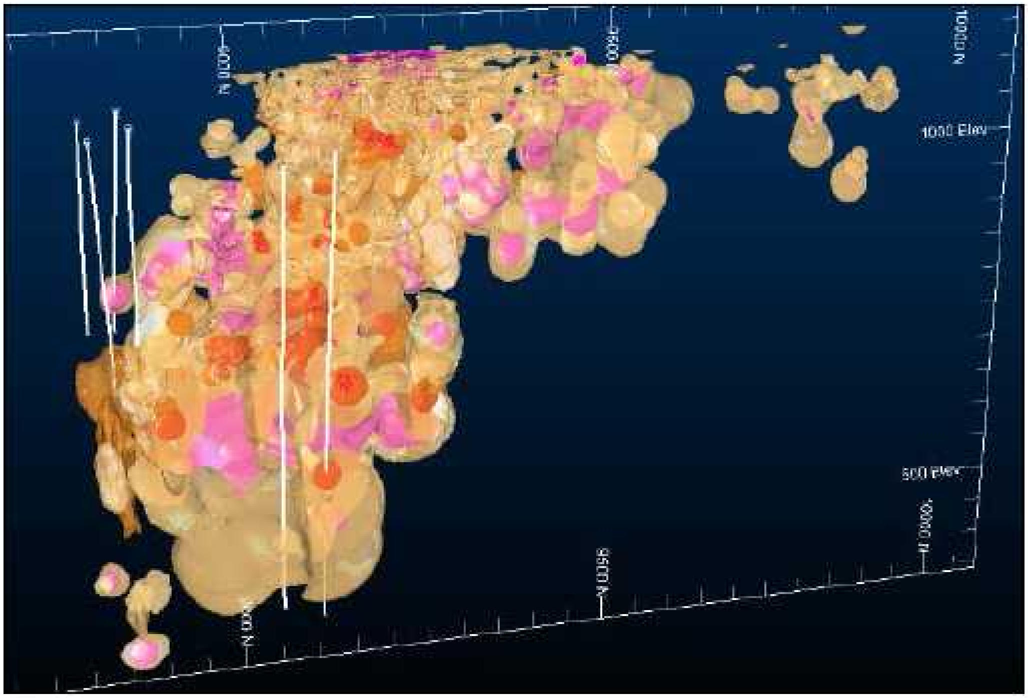


Figure 10-2: Long section looking west showing grade shells and 2011 diamond drillholes

Table 10-4: Maud Creek 2011 drilling data

Hole ID	X Collar	Y Collar	Z Collar	Depth	Type	Azimuth	Dip
MC001	19550	8799.996	1130	339	DD	275	-60.0
MC002	19500	8849.996	1130	372.6	DD	271	-59.7
MC003	19650	8799.999	1130	414	DD	279	-60.0
MC004	19600	8849.999	1130	483.6	DD	275	-60.0
MC005	19820	8974.997	1130	756.6	DD	270.9	-60.8
MC006	19770	9039.992	1130	813.5	DD	270	-60.1
MCRC001	19228.856	8539.006	1130.685	50	RC	275	-60.0
MCRC002	19281.995	8537.7	1130.949	50	RC	275	-60.0
MCRC003	19332.654	8542.016	1131.567	50	RC	275	-60.0
MCRC004	19384.094	8523.776	1132.461	50	RC	275	-60.0
MCRC005	19232.577	8725.36	1129.345	50	RC	275	-60.0
MCRC006	19282.385	8746.099	1130.508	50	RC	275	-60.0
MCRC007	19330.733	8732.725	1131.416	50	RC	275	-60.0
MCRC008	19381.4	8733.986	1131.779	50	RC	275	-60.0
MCRC009	19235.469	8937.985	1128.153	50	RC	275	-60.0
MCRC010	19286.715	8933.594	1128.934	50	RC	275	-60.0
MCRC011	19335.354	8930.858	1128.718	50	RC	275	-60.0
MCRC012	19384.542	8936.011	1128.761	50	RC	275	-60.0
MCRC013	19789.85	9342.812	1124.795	52	RC	275	-60.0

MCRC014	19210.585	10341.349	1124.459	50	RC	275	-60.0

10.1.1 Surveying

All holes were either surveyed by single shot and/or gyro survey. All drillhole collars were picked up by surveyors, and some historical drill collars were also resurveyed.

10.1.2 Core Recovery

Core recovery was not recorded for the Maud Creek 2011 drilling.

10.2 Sampling prior to 2011

Prior to the then Crocodile Gold' s acquisition, four companies had previously run exploration programs at Maud Creek, utilising both Diamond (23,004 m) and RC (48,107 m) drilling techniques (Table 10-2). A summary of the associated procedures was carried out by Snowden in their 2006 report Addendum to the Technical Report entitled Independent Technical Review of the Burnside, Union Reefs, Pine Creek and Maud Creek Gold Projects, Northern Territory, Australia - Resource Update, Maud Creek Gold Project. This report summarised any available historical data from Maud Creek produced by Placer Dome, Kalmet, Hill 50 and Terra Gold. GBS Gold requested the review by Snowden in 2006, but never conducted any drilling programs at Maud Creek during their tenure.

10.2.1 Surveying

As part of the review, GBS contracted a professional surveyor to locate all drillhole collars.

GBS also utilised the following downhole survey instruments to verify the orientation of the drillholes:

- Sperry Sun system
- Single shot system

Measurements were typically taken every 25 m.

Figure 10-3 and Figure 10-4 highlight the changes in azimuth and dip with drillhole depth as captured by GBS. The drillhole azimuths typically wander +/- 15°, with azimuth showing a relationship with depth.

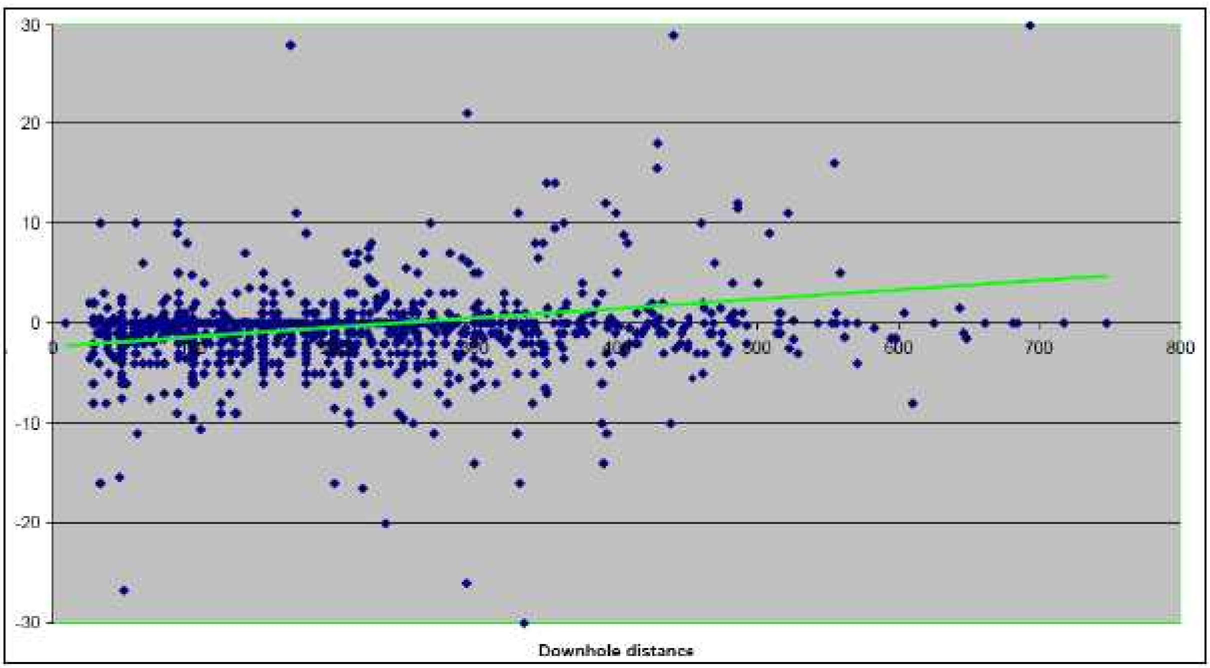


Figure 10-3: Changes in Azimuth with Depth for drilling prior to 2011

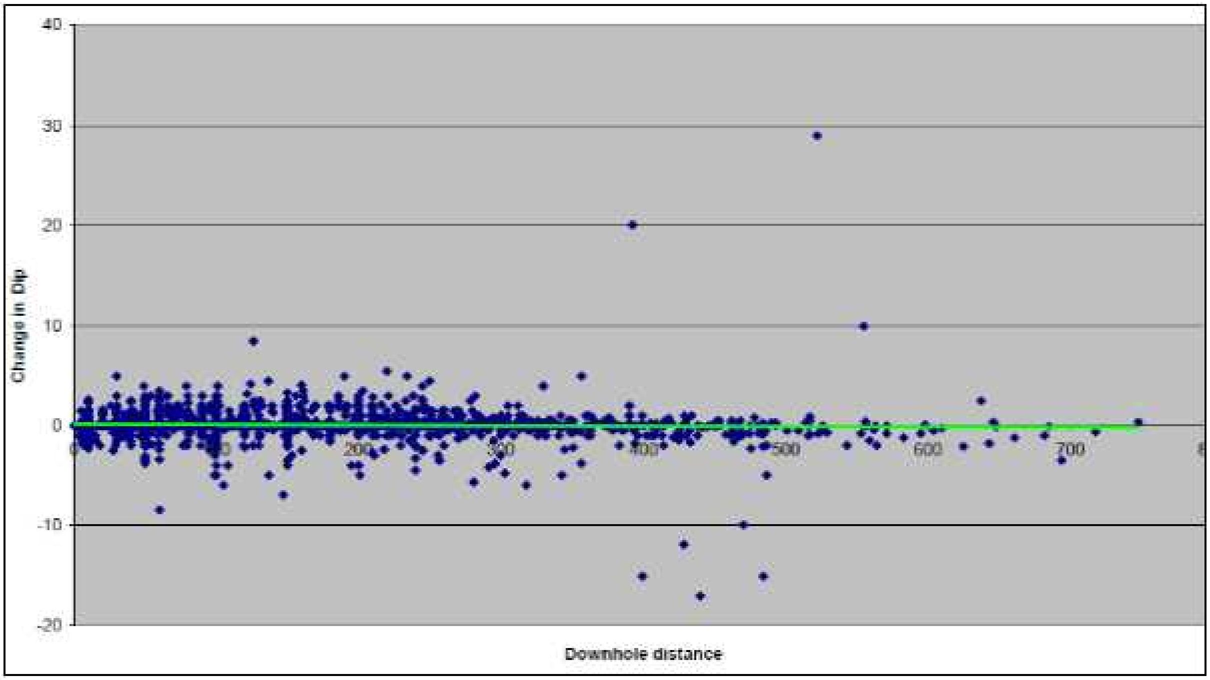


Figure 10-4: Changes in Dip with Depth for drilling prior to 2011.

Several corrections in the acQuire database were made by Newmarket Gold for typographic errors of the single shot surveys. Also, surveys not present in the database were added from source data (MC005 and MC006). Gyro surveys of MC002, MC005 and MC006 were also reprocessed, due to an incorrect application of the grid rotation.

10.2.2 Core recovery

Core recovery data is available for nine diamond holes, all of which were drilled by Hill 50 between 1997 and 2001 (Table 10-5). Overall the recorded recoveries are considered to be acceptable.

Table 10-5: Core recovery of drillholes prior to 2011

Hole ID	Average recovery%
MD045W1	100
MD045W2	97
MD053	100
MD053W1	98
MD055	98
MD055W1	97
MD055W2	96
MD055W3	98
MD056	99

Verification work completed by Newmarket Gold identified several holes which had core recovery recorded (MD017 - 19) quantitatively as 'Good'. There are some recovery records in the Database which conflict with measurements in the source data (MD014, MD024, MD029 and MD30), and geotechnical logs.

11 Sample Preparation, Analysis, and Security

11.1 Sampling Techniques

Samples used to inform the Maud Creek block model estimate are sourced from both diamond drill core and reverse circulation chip samples collected over the last 20 years.

11.1.1 Reverse Circulation Sampling for the 2011 drilling program

Figure 11-1 outlines the sampling procedure for the 2011 drilling campaign. Sample intervals (1.0 m) are washed and sieved by a geologist and then inspected to determine its geological attributes. Geology is entered directly onto standard logging sheets in either hard copy or digital form via a portable computer using standardised geological codes. Each washed sample is then stored in a chip tray and stored in a shed at the core farm for future reference.

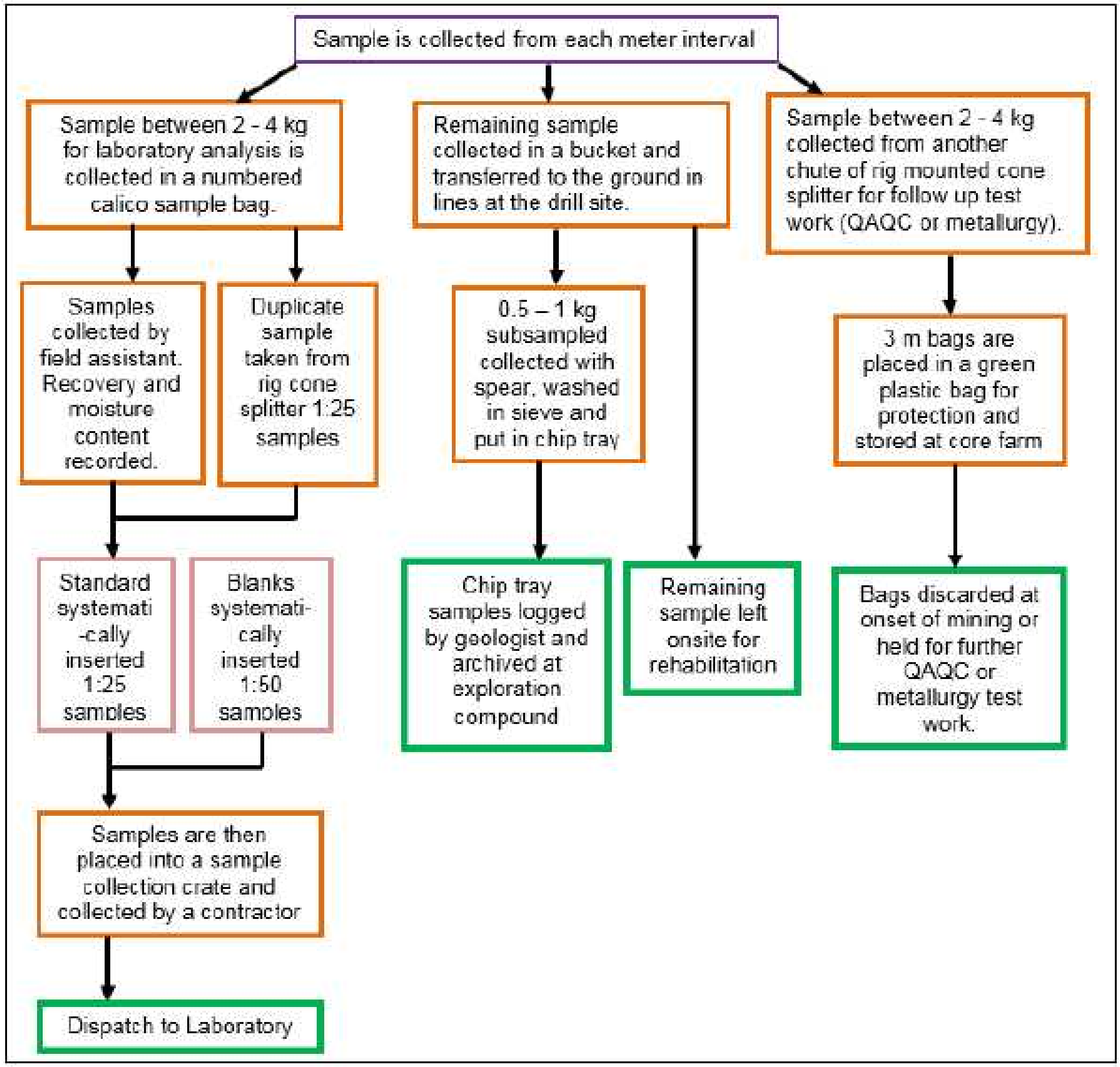


Figure 11-1: Reverse Circulation Sampling flow chart

11.1.2 Diamond Sampling for the 2011 drilling program

Much of the drill core produced from the Maud Creek area is composed of barren sediments and tuff. Therefore, not all diamond drill core is required to be sampled.

Core orientations were marked on the bottom of the core using a Camteq Orishot tool at the end of each rod. Drill core was then orientated by technicians based on these orientation marks. The Geologists then log each hole for weathering, lithology, structures, alteration, mineralization and geotechnical information. Zones of core loss are identified and marked by inserting marker blocks recording the exact length of the core loss.

At the completion of logging geologists mark the core for sampling and photograph each tray dry and wet. Samples intervals are chosen based on lithological or mineralization contacts. Sample boundaries are often made at pre-existing breaks; otherwise the half core is cut perpendicular to the core axis using an Almonte automated diamond core saw.

Minimum sample size is 0.3 m and maximum size is 1.5 m. The core was cut so as to divide the mineralization in half whilst preserving the orientation line. Some drillholes were sampled for their entire length and some were sampled from 20 - 50 m in the hanging wall through to the end of the hole.

11.1.3 Sampling prior to 2011

Details of the sample collection, preparation and quality control techniques employed by each of the previous operators of Maud Creek are not fully documented. Procedures documented in annual reports written by Kalmet Resources (1996) note that:

- Sampling techniques varied for each drilling program;
- A review of the assay database, analytical quality control and sampling techniques was undertaken in July 1996 by Geocraft Pty Ltd, an independent consultant;
- Two metre composite RC samples were collected by riffle splitting;
- 5-6 kg samples were dispatched to Alice Springs, where they were riffle split to a nominal 3kg prior to pulverizing of the entire sample;

11.2 Data Sampling and Distribution

The Maud Creek model has been shown during validation to be subject to varying drillhole density and sample locations, which has affected lode geometry. Within the upper/central parts of lodes the drilling is regular and of sufficient density, but subject to decreasing densities and irregular spacing at depth.

11.3 Testing Laboratories

11.3.1 2011 Drilling program

Assaying of the drill core and reverse circulation samples was completed by either NAL at Pine Creek, the NTEL or the ALS labs in Darwin. All laboratories used are independent of Newmarket Gold and are well known to SRK Consulting as competent assayers. Once the assaying laboratory' s personnel receive the drill or chip samples they undertake sample preparation and chemical analysis. Results are returned to Newmarket Gold staff, which validate and input the data into relevant databases.

All analytical work including sample preparation, analytical procedures, QA/QC measures and associated security and chain of custody procedures have been completed in accordance with the established protocols routinely used by Newmarket Gold.

SRK Consulting considers that these procedures and protocols are of acceptable quality and are broadly consistent with international best practice standards. Lab visits have been conducted by Newmarket Gold staff to meet with the management of the laboratories and to inspect the facilities.

11.3.2 Sampling prior to 2011

Various reports written by previous tenement holders document a range of laboratory testing or investigations. The 2006 Snowden report summarised these assessments (Table 11-1):

Table 11-1: Summary of QAQC reports written for Maud Creek

Company	Report	Year	Description
Placer	Report NT25/91	1991	202 Check-analysis by ALS Townsville of samples previously analyzed by Classic Laboratories
Kalmet	MG03/002	June 1994	Comparison of gold results in 3 sets of 'twinned' RC and Diamond Drillholes
Kalmet	KMT196 Table 4	1995	Resampling of 18 intervals in 11 holes that reported > 20 g/t Au
Kalmet	KMT196 Section 5.2.3	1996	Rectification of deliberate sample corruption in
			Laboratory (1297 samples)

11.4 Sample Preparation

11.4.1 2011 Drilling program

The following sample preparation activities (Figure 11-2) were undertaken by Newmarket Gold staff for the 2011 drilling program:

Standards, blanks, barren quartz flush and duplicates placed in pre numbered calico bags;

Sample is placed into calico bags;

Calico bags loaded into green plastic bags with the sequence of samples in the bag labelled on the outside;

The green plastic bags were then placed into dispatch cages to be picked up by courier and taken to the Laboratory.

At the completion of each hole the core trays are stored in racks for future retrieval.

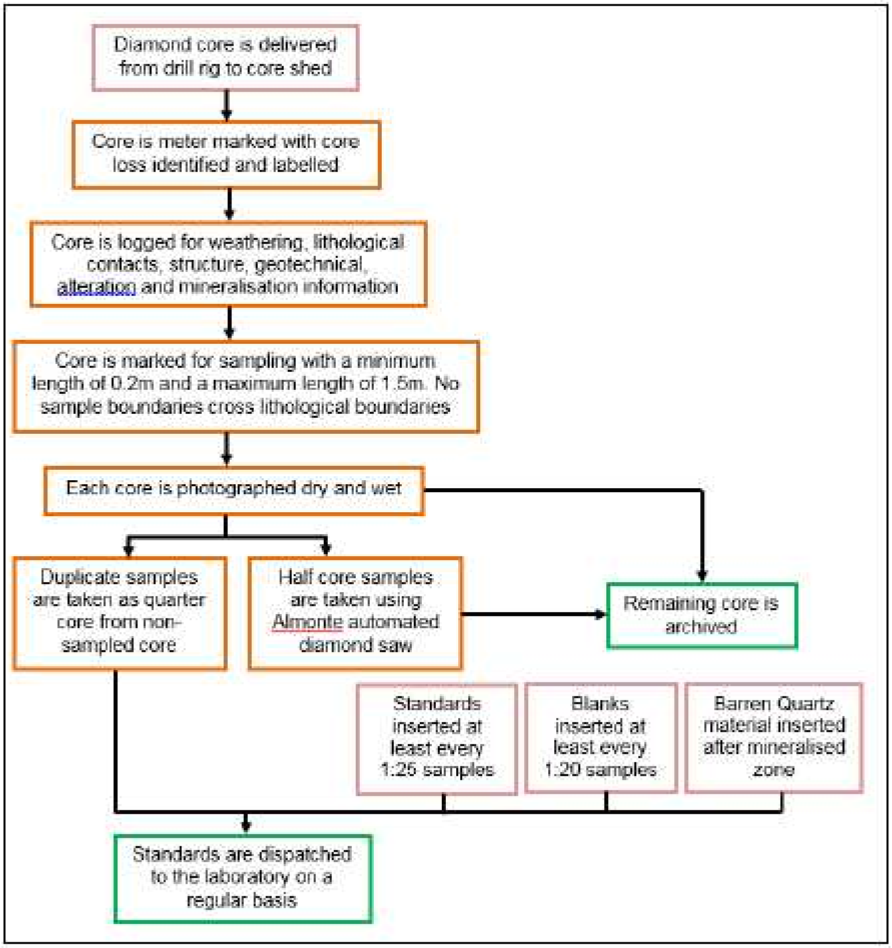


Figure 11-2: Diamond Drilling Sampling flow chart

The primary commercial laboratory used for the Maud Creek drilling campaign was Northern Territory Environmental Laboratories (NTEL), (now Genalysis), with Australian Laboratory Services (ALS) in Darwin acting as an umpire lab. Samples sent to ALS were prepared in Darwin and then sent to either the ALS laboratory facilities in Perth or Townsville for analysis.

The following sample preparation activities are undertaken by laboratory staff (Figure 11-3):

Samples are received and checked against the submission sheet;

Average sample weight for the submission is taken;

Each sample is then dried at 105°C until fully dry;

Entire sample is initially crushed in a jaw crusher to approximately 2mm;

Each sample is then rotary split with 300g taken for milling and assay and remainder set aside as a coarse reject and returned to Newmarket Gold;

The 300g sample is then milled to pass through a roll crusher to 2mm;

Samples are riffle split into two sub-samples – one is milled while the other is retained as a coarse reject and returned to Newmarket Gold;

The retained sub-sample is milled to 85% passing 75µm with 1 in 20 samples wet screened to check for compliance;

Each milled pulp samples is further split to provide 25g for fire assay (FA25) with an AAS finish and <1g used for multi-element, if required. At ALS 30g of the pulp is weighed off for fire assay with an AAS finish (AA26).

Any remained pulp sample is kept for future analysis and returned to Newmarket Gold.

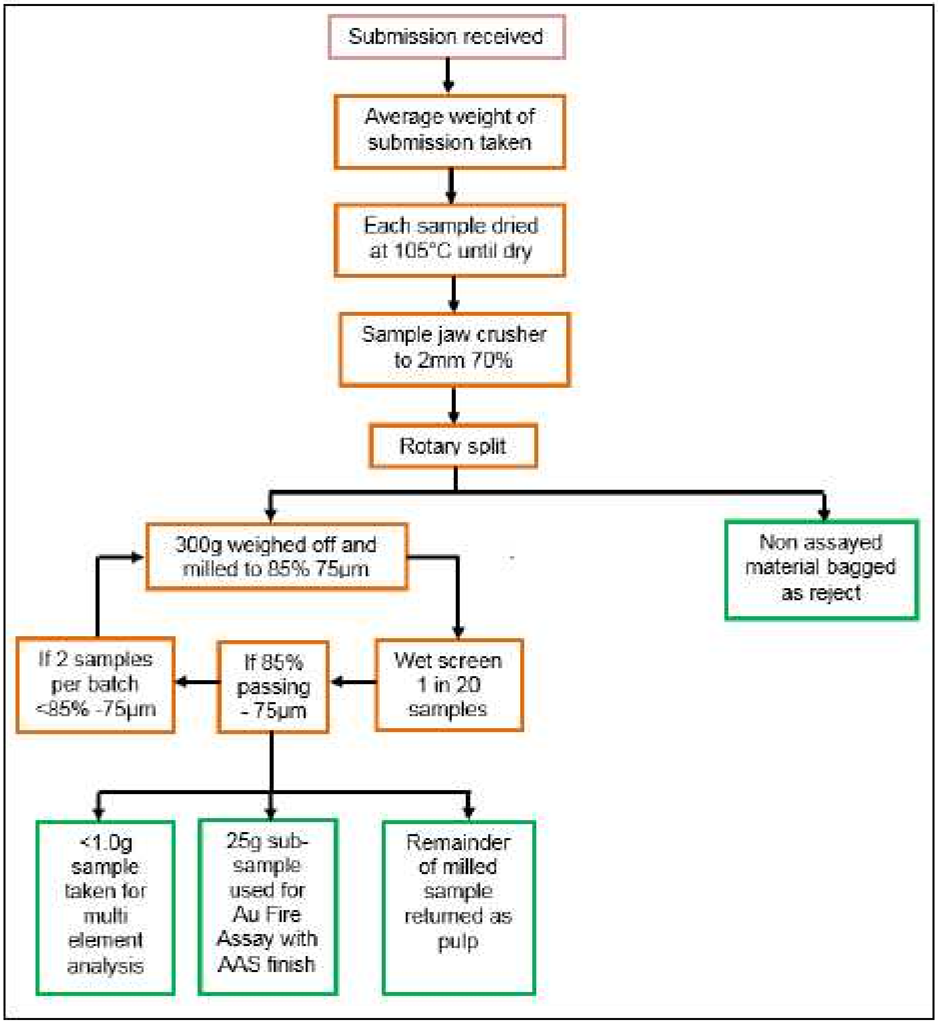


Figure 11-3: Laboratory sampling flow chart

11.4.2 Sampling prior to 2011

Multiple owners have generated the Maud Creek data set in several phases over the last 20 years. The following references were noted by Snowden in 2006:

Kalmet Resources engaged Lantana Exploration Pty Ltd in 1995 to re-enter all the analytical data obtained up to that time. This work included all drillhole data up to and including hole number MCP061;

Kalmet 1996; engaged geological consultant Geocraft Pty Ltd to verify the database and analytical procedures;

Kilkenny 1998; the MRT 1998 resource study states that data files were merged into an Access database and validated using MRT internal systems and no significant errors were detected;

Harmony 2003; a competent person has classified the Mineral Resource estimates for the Maud Creek Project in accordance with the JORC Code and these estimates have been released by Harmony to various Stock Exchanges. This chain of Competent Person Statements as required under the JORC Code has been relied upon by Terra Gold to attest to the validity of the drilling data.

Terra Gold has undertaken a validation of the database using Micromine validation routines and reported that no significant errors were detected.

11.5 Sample Analysis

11.5.1 2011 Drilling program

Maud Creek drill core and RC samples were assayed for gold and multi element analysis. Gold grades are determined by fire assay/ atomic absorption spectroscopy (AAS) and multi element (silver, arsenic, bismuth, calcium, cobalt, chromium, copper, iron, potassium, manganese, molybdenum, sodium, nickel, lead, antimony, tin, titanium, zinc, zirconium) by ICP Atomic Emission Spectrometry. The following procedure is undertaken:

50 g of pulp is fused with 180 g of flux (silver);

Slag is removed from the lead button and cupellation is used to produce a gold/ silver prill;

0.6 mL of 50% nitric acid is added to a test tube containing prill, and the test tube is placed in a boiling water bath (100°C) until fumes cease and silver appears to be completely dissolved;

1.4 mL of hydrochloric acid (HCl) is added;

On complete dissolution of gold, 8 mL of water is added once the solution is cooled; and

Once the solids have settled, the gold content is determined by fire assay/atomic absorption AAS.

The following procedure is undertaken for multi element analysis:

A 0.25g sample is pre- digested for 10-15 minutes in a mixture of nitric and perchloric acids;

Hydrofluoric acid is added and the mixture is evaporated to dense fumes of perchloric;

Residue is leached in a mixture of nitric and hydrochloric acids;

Solution is then cooled and diluted to a final volume of 12.5mls;

Elemental concentrations are measured simultaneously by ICP Atomic Emission Spectrometry.

11.5.2 Sampling prior to 2011

Kalmet utilised ALS in Alice Springs for gold and arsenic assays;

All samples were analyzed by ALS in Alice Springs for gold by fire assay method PM209 and for arsenic by AAS method G003 or G102;

Selected holes as MD21 to MD31 were assayed for copper, lead, zinc, silver, nickel, antimony, bismuth, chromium by AAS method G102. Samples from MD21 were also analyzed for Hg by AAS method G008;

Check samples for gold were analyzed by Analabs in Townsville;

Duplicate check samples were assayed for gold by either ALS in Alice Springs or Assay Corp in Pine Creek;

SG measurements were completed on 22 oxide and transition mineralization and wall rocks, selected from MD15 to MD19. Measurements were completed by Assay Corp in Pine Creek.

11.6 Laboratory Reviews

The two laboratories used by Newmarket Gold for Maud Creek offer different preparation techniques with the 25 g fire assay by NTEL and a 30 g fire assay by ALS. The following summarises findings with respect to assay work from the two independent laboratories:

There are some errors in the datasets. Typographical errors, wrong standards, recorded/sent to the lab, obvious swaps in the databases. These errors are collaborative from both the laboratories and the database operator;

ALS report lab standards while NTEL do not;

Outright errors should not be appearing in the database (standards and replicates);

Proper control charting methods should be applied to fire assay batches that indicate standards outside proper control limits;

The lack of blanks inserted prior to sample submission needs to be addressed;

It is of the opinion of SRK that the sampling preparation, analysis and security procedures are all adequate for use in these Mineral Resource and reserve estimates.

NTEL is an independent laboratory based in Darwin. The relationship between NTEL and Newmarket Gold is on a client/ supplier arrangement with a contract in place for service. ALS laboratories are certified using the ISO9001:2008 accreditation (Quality Management Systems - Requirements). They also hold the NATA Technical accreditation under ISO17025:2005.

11.7 Assay Quality Assurance and Quality Control

11.7.1 Standard Reference Material

2011 Newmarket Gold drilling program:

Certified standards are submitted to the laboratory on a regular basis. A standard is inserted into every batch every 117 samples during the RC program and every 26 samples during the diamond drilling program (or less).

There were initially 3 standards across all ranges used during the RC drilling program and 5 standards across all ranges used during the diamond drilling program (Table 11-2).

Each standard for each drill type is charted chronologically to check for compliance and any progressive trends, which may be apparent. An example of the chart used to chronologically check the standards is presented in Figure 11-4.

A total of 48 standards were used against the 1243 samples taken for the diamond drilling program with 6 standards inserted for the 699 samples taken for the RC program. No laboratory standards were inserted.

Table 11-2: Standard ST202/5355 Compliance table for 2011 drilling program

Standard	ST202/5355
Recommended value	2.37
Mean Result	2.35
AUD% difference versus RV	-1.0
Standard Deviation	0.07
Number of assays	17
Number > -2SD	0
Number > +2SD	0
% +/- 2SD	100

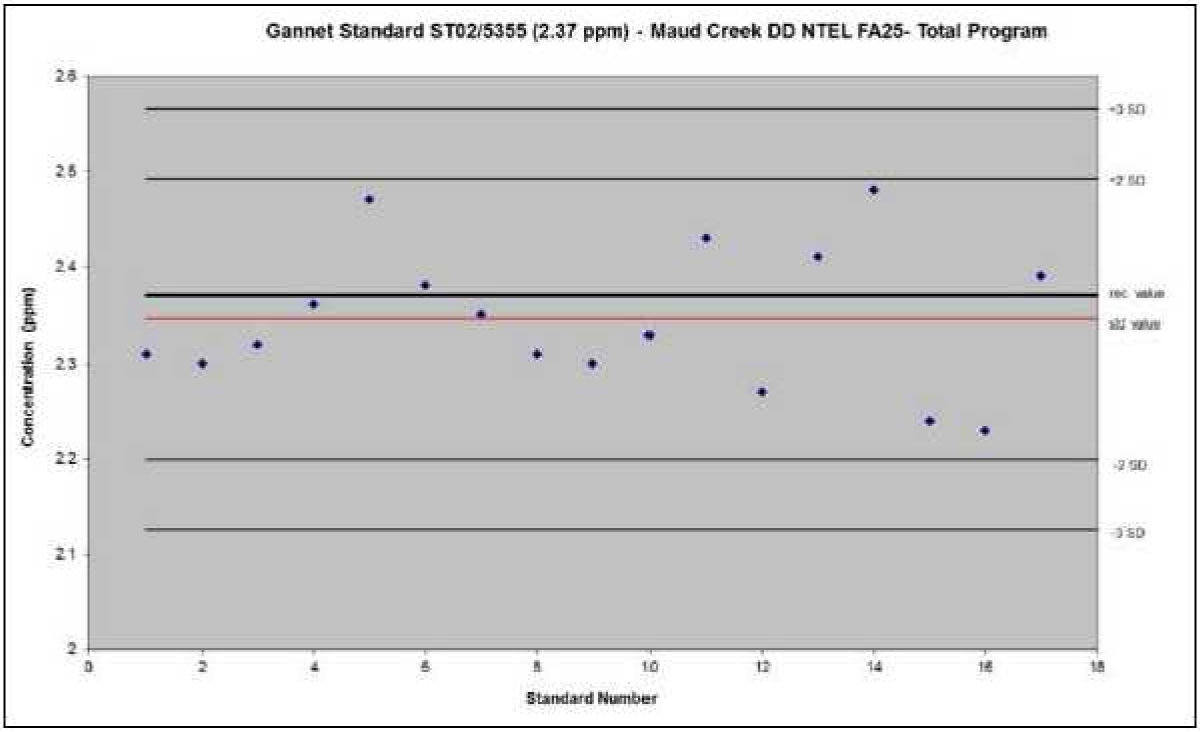


Figure 11-4: Standard ST202/5355 Compliance chart for 2011 drilling program

Sampling prior to 2011

The Maud Creek digital database notes that four certified standards were inserted during the Maud Creek drilling program run by Placer between 1990 and 1991 (Table 11-3).

Table 11-3: Maud Creek Certified Laboratory Standards

Standard	Grade (g/t Au)
OxE20	0.548
OxG22	1.035
OxH19	1.344
OxH29	1.298

Figure 11-5 to Figure 11-8 are plots sourced from pre 2011 drill data, and show difference (%) and absolute difference (%) for each standard assay on the left-hand vertical axis, with gold grade (g/t) on the right-hand vertical axis. The standard value is plotted in purple, and the CRM lab value is plotted in blue. Values of greater than 20% difference between certified value and the determined value appear to be related to submission error or incorrect standard.

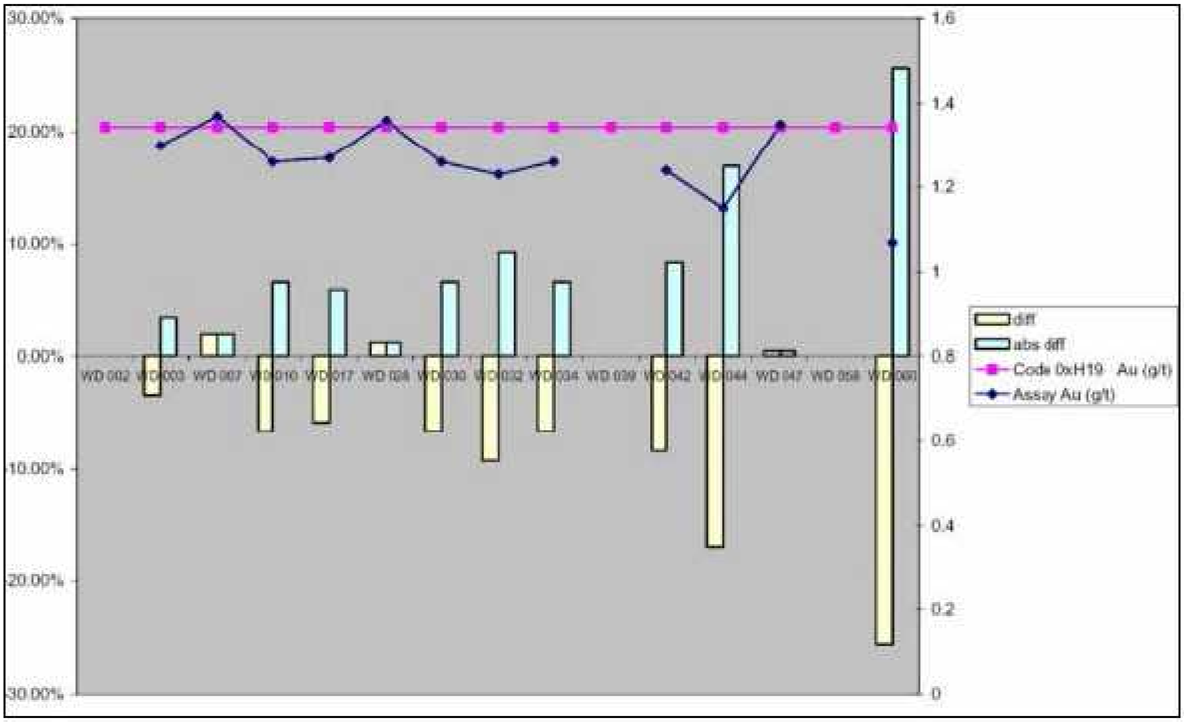


Figure 11-5: Standard OxH19

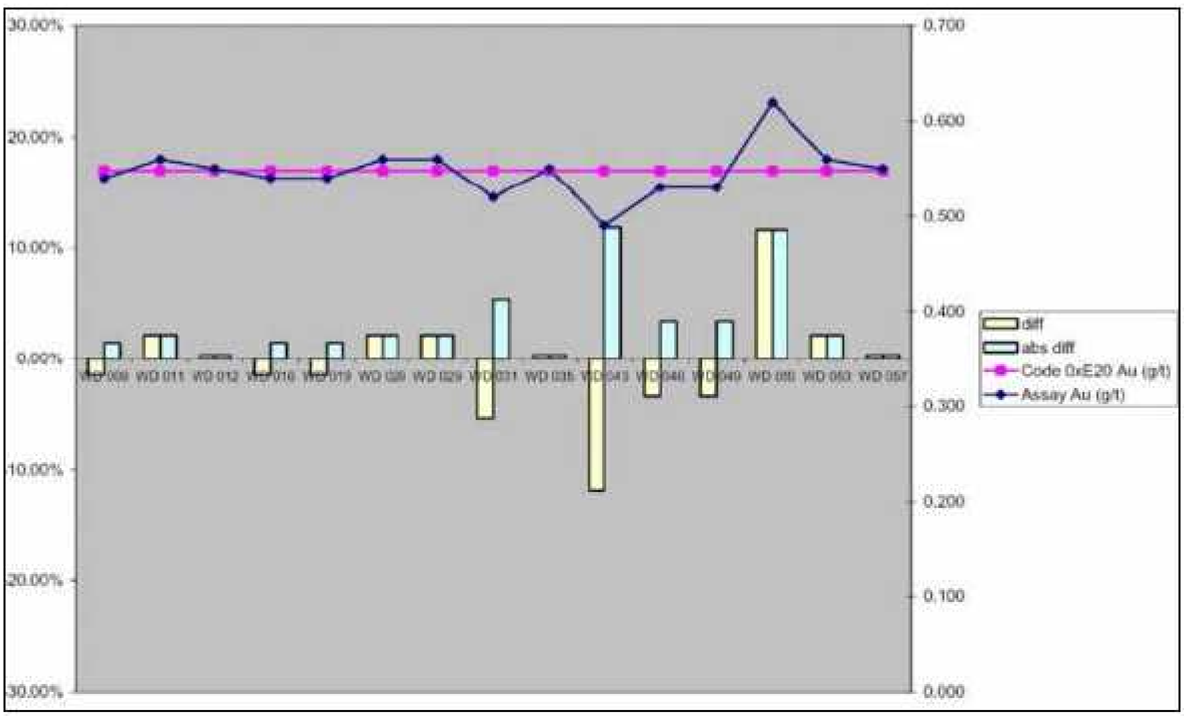


Figure 11-6: Standard OxE20

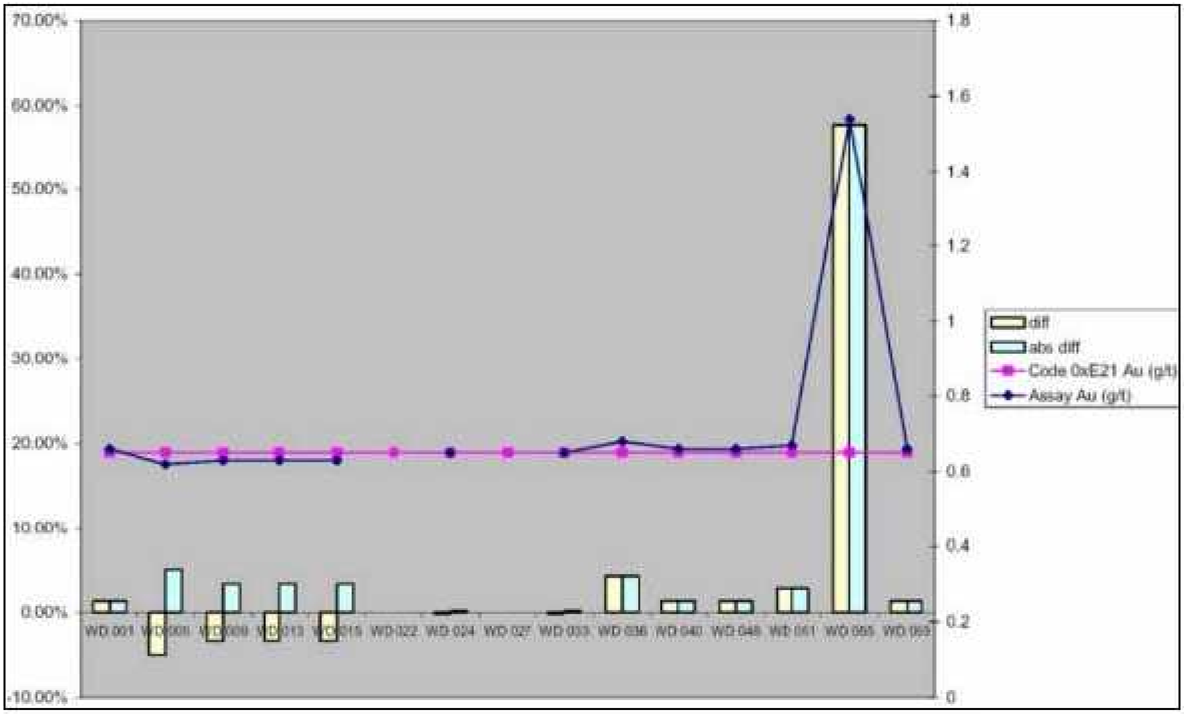


Figure 11-7: Standard OxE21

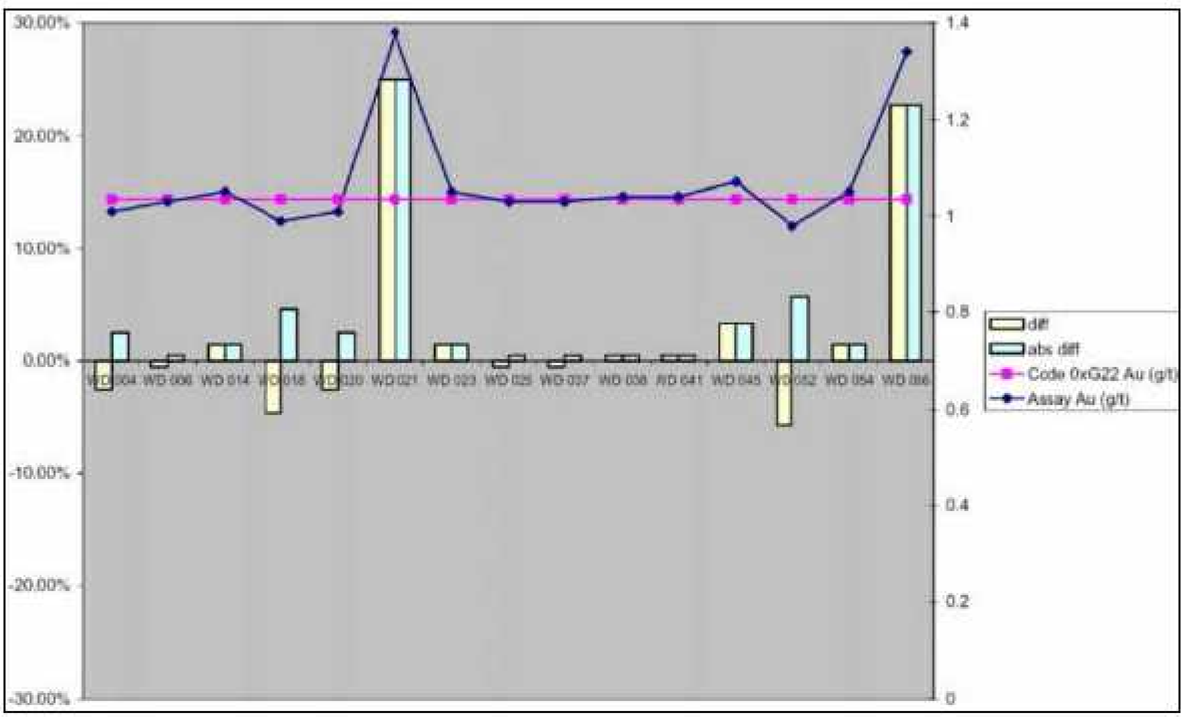


Figure 11-8: Standard OxG22

11.7.2 Blank Material

2011 Newmarket Gold drilling program:

Blank materials included in the sample stream were derived from several sources; barren core, barren coarse rejects, crushed Bunbury Basalt (from Gannet Holding Pty Ltd, referred to in this report as blank). Blank results above 0.02 g/t Au are queried and any issues resolved. Results are chronologically charted to visually check compliance (Figure 11-9). No blanks were inserted into the Maud Creek RC drilling program. For the diamond drilling program, a total of 72 blanks were inserted with 98.6% at or below 0.02 g/t Au.

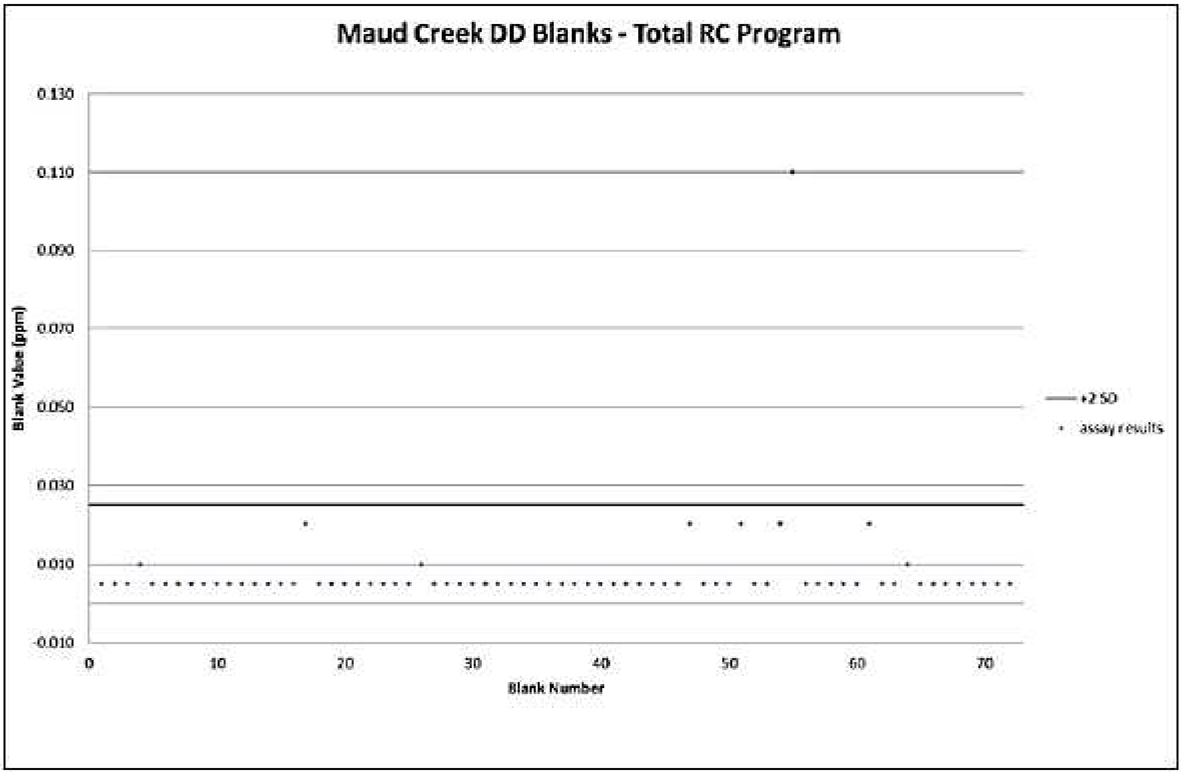


Figure 11-9: Compliance chart for blanks used in the 2011 Maud Creek drilling program.

Sampling prior to 2011

There was no evidence in any reporting indicating the insertion of blank material with samples submitted to laboratories.

11.7.3 Duplicate Assay Statistics

2011 Newmarket Gold drilling program:

Relative precisions have been used to analyze the precision of duplicate samples. The relative precision is a measure of dissimilarity, that is, if both distributions are exactly the same, this value will equal zero increases as the distributions become more dissimilar.

In this report, relative precision has been calculated using all data pairs for the ranges of below detection (<0.01 g/t) to 0.20 g/t Au, 0.21 to 0.5 g/t Au, 0.51 to 0.7 g/t Au, 0.71 to 1.00 g/t Au, 1.01 to 1.40 g/t Au, 1.41 to 5.00 g/t Au and >5.00 g/t Au. This is to isolate the large conditional variance of errors associated with assay determinations near both lower and upper analytical detection limits and to selectively analyze results within these set ranges.

An example of the analysis tables for the 2011 Maud Creek drill program is given in Table 11-4 to Table 11-6 and Figure 11-10.

Table 11-4: Duplicate analysis table for the 2011 drilling program

Repeat	Maud Creek DD NTEL FA25 Total Program
Mean original results:	0.06
Mean repeat results:	0.07
Number of assays:	39
Standard Deviation:	0.07
Sum of Differences:	-0.39
Sum of Diff * Diff:	0.18
Mean Difference:	-0.01
% Results within +/- 2SD	102
Results within 30% precision level	86
Average absolute% Difference:	24
% Assays original <or = repeat	83

Table 11-5: Duplicate correlation table for the 2011 drilling program

Range (g/t)	Original vs Repeat
Combined	0.952
<0.20	0.777
0.21 - 0.50	-
0.51 - 0.70	-
0.71 - 1.00	1.000
1.01 - 1.40	-
1.41 - 5.00	-
>5.01	-

Table 11-6: Duplication R Table for the 2011 drilling program

Range (g/t)	# of assays	% of total #	Mean original	Mean repeat	% diff between means (bias)	Average% diff between assays (bias)	Absolute average% diff between assays (total error)	Standard Deviation
<0.20	38	97	0.03	0.04	-21.8	-9.00	24	0.07
0.21 - 0.50	0	0	-	-	-	-	-	-
0.51 - 0.70	0	0	-	-	-	-	-	-
0.71 - 1.00	1	3	1.08	1.19	-10.2	-10.19	10	0.00
1.01 - 1.40	0	0	-	-	-	-	-	-
1.41 - 5.00	0	0	-	-	-	-	-	-
>5.01	0	0	-	-	-	-	-	-
Total	**39**	**100**	**0.06**	**0.07**	**-16.5**	**-1.3**	**24**	**0.07**

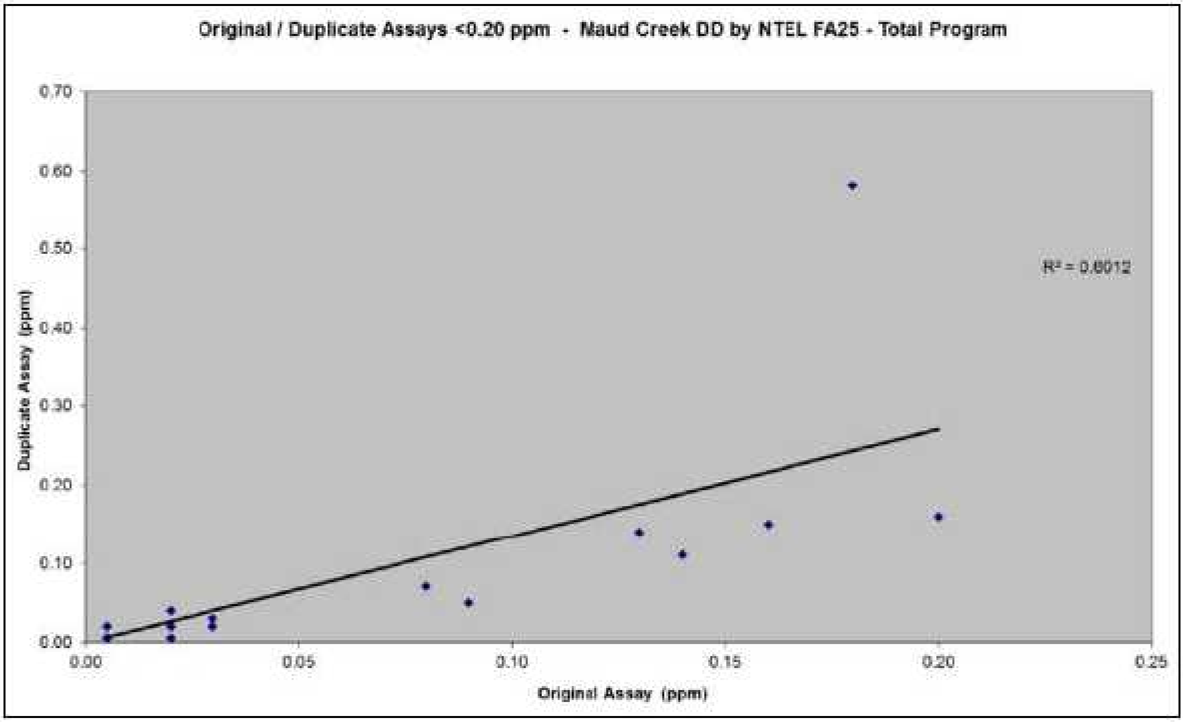


Figure 11-10: Duplicate correlation plot; range <0.2 g/t for 2011 drilling program

Eighty Six per cent of diamond duplicates for Maud Creek fall within the 30% precision level. All but one of the 39 original samples is above 0.2 g/t Au with 24 original samples being below the detection limit. Of the 29 RC duplicates taken, 24 of the original samples are below the detection limit with 87% falling within the 30% precision level.

Sampling prior to 2011

The database for pre 2011 drilling contains laboratory repeat data (Au1 and Au2). Figure 11-11 is a relative difference plot of the two data sets. It would generally be expected that as the average grade of each pair of data increases the relative difference between the paired data would decrease. The plot does not indicate this, suggesting some issues with laboratory precision. This may reflect the presence of coarse or nuggety gold. It is understood that no screened fire assay analysis were undertaken to help assess whether coarse gold is an issue at Maud Creek. Furthermore it must be noted that these are (presumably) repeats initiated by the laboratory, and not blind submissions of field duplicates, which may be normally expected to show poorer precision than laboratory repeats.

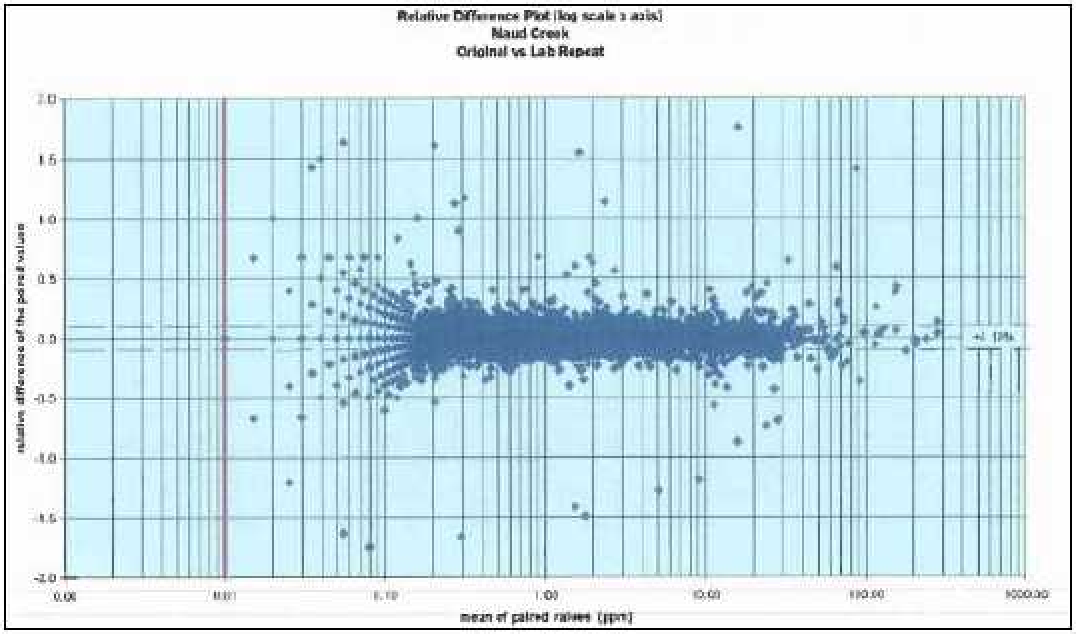


Figure 11-11: Relative difference plot original vs lab repeats for pre 2011 Maud Creek drill data

11.7.4 Internal laboratory Repeats

2011 Newmarket Gold drilling program:

Internal laboratory repeats were taken for both RC and diamond drilling at the primary laboratory (NTEL). Relative precisions have been used to analyze the precision of repeat samples. The relative precision is a measure of dissimilarity, that is, if both distributions are exactly the same, this value will equal zero increases as the distributions become more dissimilar.

In this report, relative precision has been calculated using all data pairs for the ranges of below detection (<0.01g/t) to 0.20,g/t Au,0.21 to 0.5g/t Au, 0.51 to 0.7g/t Au, 0.71 to 1.00g/t Au, 1.01 to 1.40g/t Au, 1.41 to 5.00g/t Au and >5.00g/t Au. This is to isolate the large conditional variance of errors associated with assay determinations near both lower and upper analytical detection limits and to selectively analyze results within these set ranges.

An example of the analysis tables for the 2011 Maud Creek drill program is given in Table 11-7 to Table 11-9 and Figure 11-12.

Table 11-7: Repeat analysis table for the 2011 drilling program:

Repeat	Maud Creek DD NTEL FA25 Total Program
Mean original results:	0.36
Mean repeat results:	0.35
Number of assays:	356
Standard Deviation:	0.16
Sum of Differences:	2.75
Sum of Diff * Diff:	9.55
Mean Difference:	0.01
% Results within +/- 2SD	98
Results within 30% precision level	96
Average absolute% Difference:	1
% Assays original <or = repeat	97

Table 11-8: Repeat correlation table for the 2011 drilling program

Range (g/t)	Original vs Repeat
Combined	0.991
<0.20	0.999
0.21 - 0.50	0.993
0.51 - 0.70	0.940
0.71 - 1.00	1.000
1.01 - 1.40	0.998
1.41 - 5.00	0.976
>5.01	0.998

Table 11-9: Repeat R Table for the 2011 drilling program

Range (g/t)	# of assays	% of total #	Mean original	Mean repeat	% diff between means (bias)	Average % diff between assays (bias)	Absolute average% diff between assays (total error)	Standard Deviation
<0.20	271	76	0.02	0.02	-0.5	-0.1	0	0.00
0.21 - 0.50	34	10	0.34	0.34	0.3	0.5	1	0.01
0.51 - 0.70	7	2	0.59	0.60	-2.2	-1.9	2	0.04
0.71 - 1.00	10	3	0.83	0.83	0.0	0.0	0	0.00
1.01 - 1.40	8	2	1.14	1.14	-0.2	-0.2	0	0.01
1.41 - 5.00	23	6	2.74	2.65	2.3	2.3	6	0.25
>5.01	3	1	9.13	8.87	13.5	13.5	21	2.01
Total	**356**	**100**	**0.36**	**0.35**	**2.1**	**0.1**	**1**	**0.16**

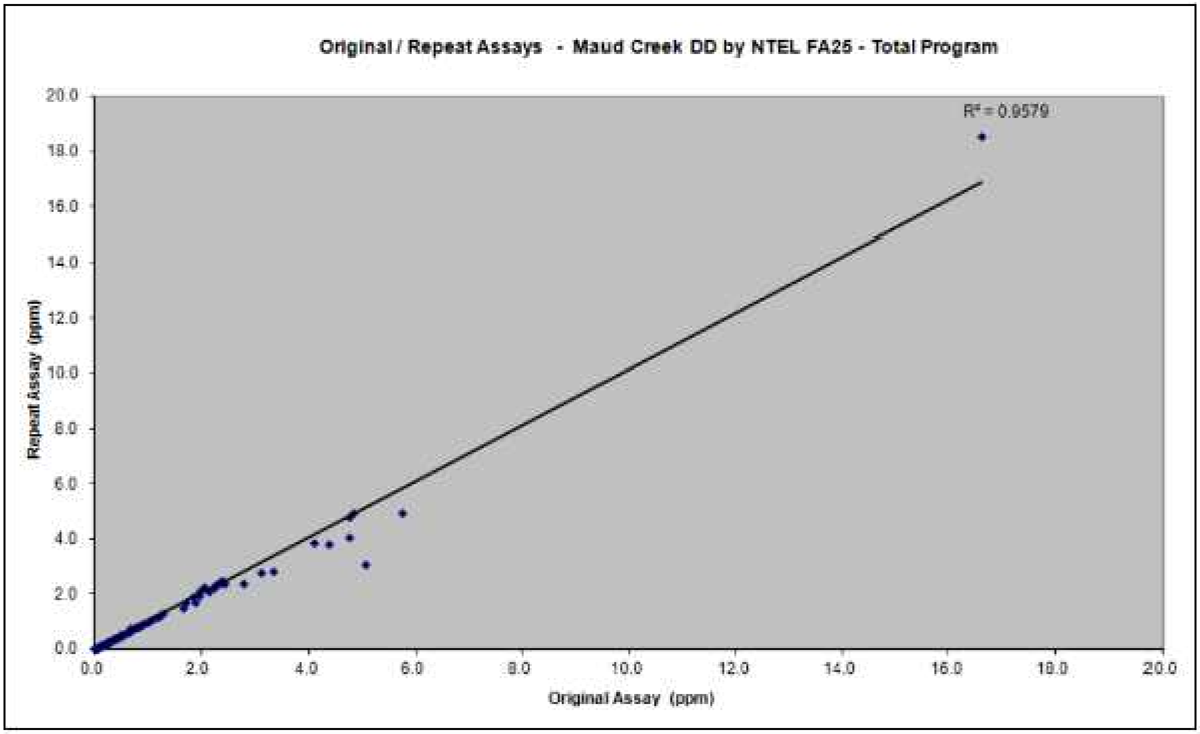


Figure 11-12: Repeat correlation plot; range <20 g/t for 2011 drilling program

Ninety six per cent of diamond repeats for Maud Creek fall within the 10% precision level. One hundred and ninety seven of the original samples are below the detection limit. Eighty five of the 356 original samples are above 0.2 g/t Au. Of the 7 RC repeats taken, none of the original samples are below the detection limit with 26% falling within the 10% precision level.

11.7.5 Inter-laboratory Repeats

2011 Newmarket Gold drilling program:

An example of the analysis tables for the 2011 Maud Creek drill program is given in Table 11-10 to Table 11-12 and Figure 11-13.

Table 11-10: Inter-laboratory repeat analysis table NTEL: ALS for the 2011 drilling program

Repeat	Maud Creek DD NTEL:ALS Total Program
Mean original results:	1.52
Mean repeat results:	1.42
Number of assays:	77
Standard Deviation:	0.35
Sum of Differences:	7.75
Sum of Diff * Diff:	9.18
Mean Difference:	0.10
% Results within +/- 2SD	95
Results within 30% precision level	73
Average absolute% Difference:	7
% Assays original <or = repeat	44

Table 11-11: Inter-laboratory repeat correlation table NTEL: ALS for the 2011 drilling program

Range (g/t)	Original vs Repeat
Combined	0.990
<0.20	0.953
0.21 - 0.50	0.964
0.51 - 0.70	0.109
0.71 - 1.00	0.743
1.01 - 1.40	0.885
1.41 - 5.00	0.907
>5.01	0.998

Table 11-12: NTEL: ALS Inter-laboratory repeat R Table for the 2011 drilling program

Range (g/t)	# of assays	% of total #	Mean original	Mean repeat	% diff between means (bias)	Average% diff between assays (bias)	Absolute average% diff between assays (total error)	Standard Deviation
<0.20	14	17	0.11	0.10	5.8	6.24	11	0.02
0.21 - 0.50	21	26	0.40	0.39	2.8	2.87	5	0.03
0.51 - 0.70	6	7	0.63	0.66	-4 .2	-4.26	7	0.05
0.71 - 1.00	7	9	0.84	0.84	0.0	0.06	4	0.05
1.01 - 1.40	6	7	1.13	1.11	1.3	1.11	4	0.06
1.41 - 5.00	24	29	2.64	2.36	10.5	7.44	10	0.61
>5.01	4	5	8.34	8.09	2.9	3.71	4	0.45
Total	**82**	**100**	**1.52**	**1.52**	**0.0**	**1.1**	**7**	**0.34**

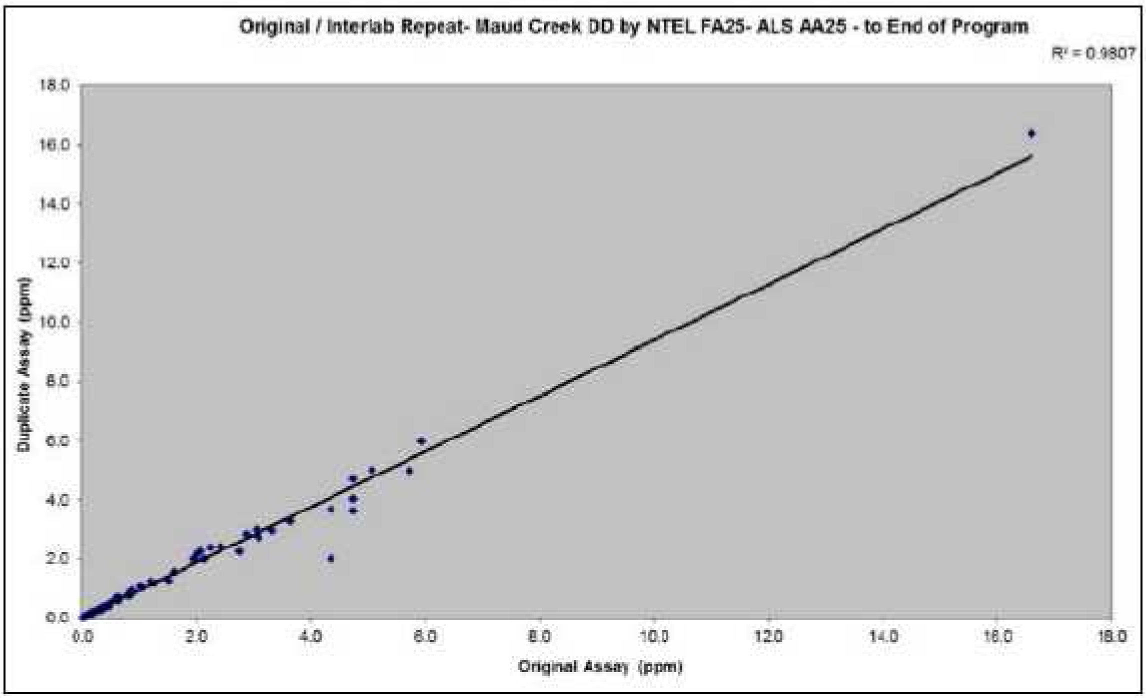


Figure 11-13: NTEL: ALS Inter-laboratory repeats for all ranges NTEL FA25: ALS AA25 for the 2011 drilling program

Seventy three percent of all diamond pulp samples fall within the 10% precision level for inter laboratory repeats. The limited population of inter laboratory repeats for both the diamond and RC programs limits this data. Seventeen percent of all RC pulp samples fall within the 10% precision level for inter laboratory repeats however only 2 samples are above 0.2 g/t with samples approaching the lower detection limit significantly affecting the precision results.

For the 2011 Newmarket Gold soil sampling program:

Soil sampling programs were undertaken at Maud Creek. 2488 soil samples were taken at Maud Creek with 82 (3.2%) of them being duplicate samples. Samples were sent to ALS in Perth and analyzed using their Ionic Leach MEMS 23 method. Eighty four percent of duplicate samples taken from Maud Creek were within the 30% precision level.

11.7.6 Sampling prior to 2011

Some umpire analysis was undertaken between ALS (the primary laboratory) and Assay Corp laboratories. It is not known whether any common CRM was submitted to both laboratories to assist in calibrating the results. Figure 11-14 is a log Q-Q plot of the ALS data against the Assay Corp data and suggests that ALS is slightly under-reporting relative to Assay Corp.

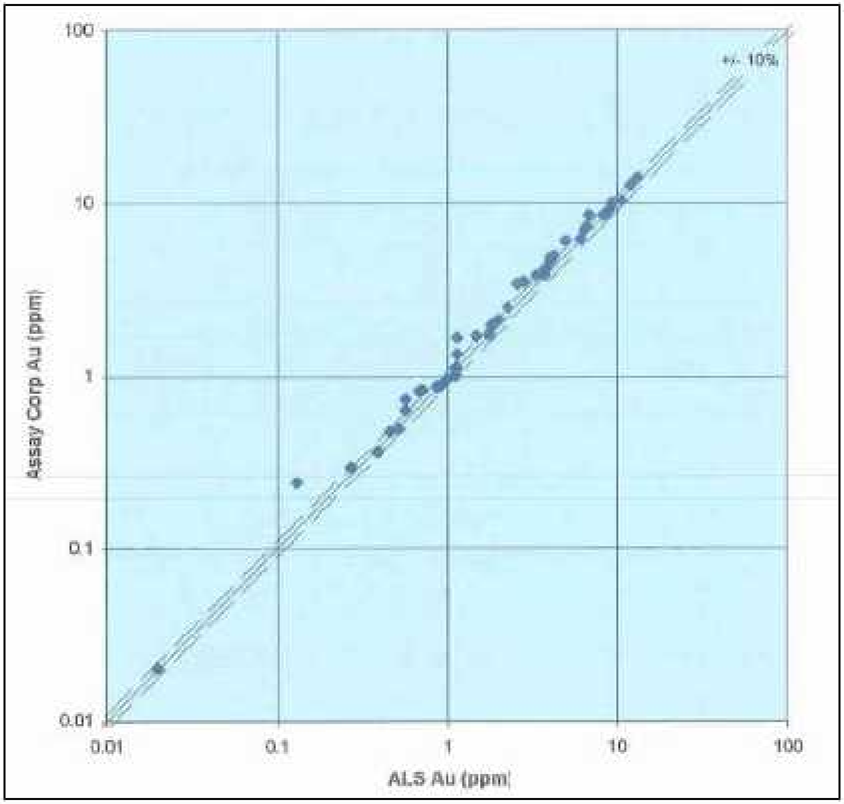


Figure 11-14: ALS: Assay Corp Inter-laboratory check analysis for pre 2011 Maud Creek drilling

Some samples were re-split, re-assayed and compared against the original data. Figure 11-15 is a log scatter plot of the re-split data which suggests a slight bias towards the original assay, although the overall correlation is acceptable. However, as the bulk of the data grades are less than 1 g/t Au, the results have little relevance to the Maud Creek estimate.

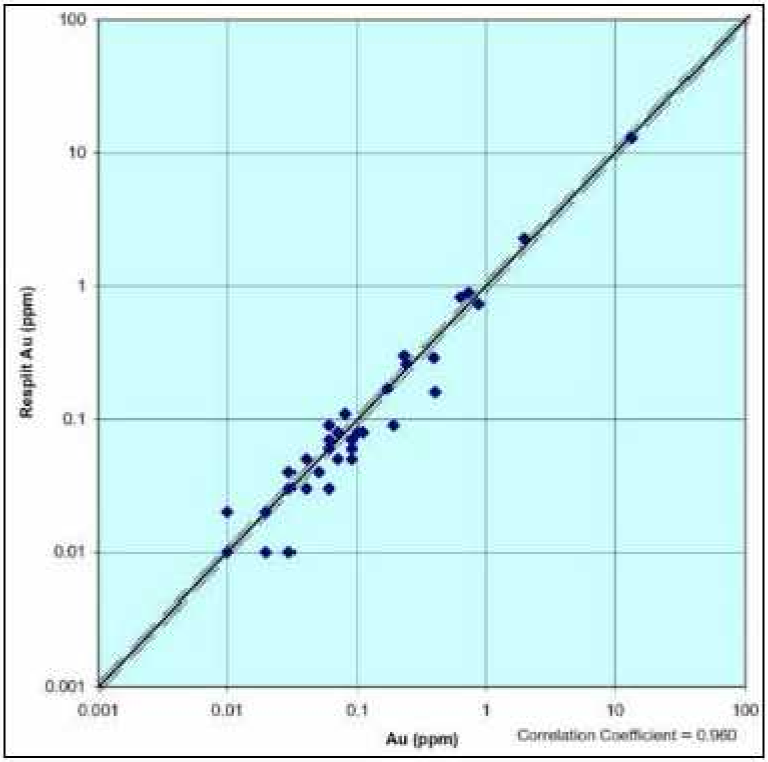


Figure 11-15: ALS Re-split check assays for pre 2011 Maud Creek drilling

11.8 Sample Transport and Security

11.8.1 2011 Newmarket Gold drilling program

A Newmarket Gold staff member is stationed on the RC drill rig while samples are being drilled and collected. At the end of shift samples were generally transported to the sample collection area where they are stored in crates as they await transportation the lab. Samples are shipped in regular intervals so they are not in crates for a length of time. These samples are located at the Brocks Creek exploration office, which can be secured if no staff member is on site.

In terms of diamond drilling, the core is collected daily from the rig and transported to the exploration office near the old Brocks Creek underground mine. Prior to the samples being transported, a photo was taken on site. This was to ensure there was a record of the material drilled before it left site, it also served the purpose of having a geological record of the drilling in case the core was damaged during transportation. The drill core is then stored in the core shed for logging and sampling. The core shed is located in a compound with security fencing. This location is locked up when no Newmarket Gold staff member is on site. Samples are cut at this location and loaded into lab crates once in calico sample bags as they await collection. These samples are then transported directly to the lab for analysis.

Once assaying is complete the results are returned in digital format to the data entry personnel employed by Newmarket Gold. These files are then loaded directly into a Datashed database. Validation via a visual comparison of standard and blanks against received values. Any questionable results are then raised with the laboratory and resolved. Submissions outside given QAQC guidelines are rejected and not loaded until resolved by the laboratory. The Datashed database is located at the Exploration office and the software is a SQL database with built-in security limiting access to people outside the Company network.

11.8.2 Sampling prior to 2011

After taking custody of the drill core, Geologists' conducted an industry compliant program of geological logging, photography, density measurements, and core sampling. Core was logged in detail onto paper and then entered into the project database. A site visit was completed in January 2006 by Snowden and the drill core was found to be well handled and maintained.

11.9 Conclusions

11.9.1 2011 Newmarket Gold drilling program

The results from the QAQC analysis of drilling has indicated a good level of confidence in assay grades for use in the resource model. The following recommendations for improvements in the current procedures are:

An immediate follow up with the laboratory when controls fail;

Increase in the regularity of blank material within the sample stream with 1:50 for RC drilling and 1:20 for diamond drilling;

An increase in the regularity of standards inserted to the desired 1:25 rate;

Inter laboratory repeats to meet or exceed a rate of 1:20 to original samples;

Assay results to be thoroughly assessed for errors prior to loading;

Conducting an analysis on barren core that is re used to serve as blanks for future batches; and

Regular tracking of QAQC compliance.

11.9.2 Sampling prior to 2011

Recommendations were made by Snowden in 2006 to improve the sampling procedures:

The current QA/ QC programs should be continued for all future sample programs at Maud Creek;

Continuation of the compilation and documentation of historical work undertaken at Maud Creek;

Systematic analysis and reporting of the QA/QC data acquired during sampling; and

Regular auditing of the database and sampling procedures in order to maintain the integrity of the database.

Since Newmarket Gold has taken ownership of Maud Creek, the recommendations made by Snowden in 2006 have been incorporated into sampling procedures.

12 Data Verification

A thorough examination of all available information was conducted by SRK and Lee Beer of Newmarket Gold. A summary of the issues identified can be found below:

- Several elements from historical assaying had not been imported into the database, notably sulphur and arsenic
- Differences in end of hole depths between the Datashed and acquire databases
- Grid conversion issues between regional MGA and calculated local grid coordinates
- Lack of clarity when distinguishing between original and re- drilled holes (same collar coordinates but differing end of hole depths)
- Issues with gyroscopic surveys due to the magnetic correction being erroneously applied.
- Missing assays from the Datashed database
- Missing assays from original source (pdf document)
- Different assay values between Datashed and acQuire databases
- 'Self-referencing' field duplicates
- Conflicting core recovery values between Datashed and acQuire databases
- Overlapping intervals in geology logging
- Interval gaps in geology logging
- Unrecognized logging codes
- Logged intervals beyond end of hole depth
- Different types of values logged for the same variable; for example, RQD logged at both a percentage and a metre value
- Missing geology logs from either digital database; only present in scanned pdf document

All issues have either been rectified by Newmarket Gold/ SRK or were deemed immaterial for the current resource estimate and will be entered/corrected in the Datashed database when time permits. SRK also conducted a site visit to verify the logging codes used in the 2011 Newmarket Gold drilling, and any available historic drilling. SRK believes the level of geological logging utilised throughout the Maud Creek drilling programs is sufficiently consistent and representative to use for a Mineral Resource Estimate.

All available QAQC reports were analyzed by SRK, to ensure the sample preparation and analysis conducted for each drill program was consistent with industry standards (Table 12-1). QAQC reports prior to 1998 did not include any analysis of the CRM's or blanks, but did include an investigation of field duplicates. Check laboratories have been utilised throughout the Maud Creek drilling programs, generally between ALS, Assay Corp and NAL. Generally QAQC reports were created by external consultants, either GEOCraft or Snowden, and concluded that the sample preparation and analysis techniques used had been appropriate and consistent with industry standards.

Table 12-1: Summary of QAQC Reports completed for Maud Creek

Company	Year	Program	Laboratory	Check Lab	CRM analysis checked	Blank analysis checked	Duplicate analysis checked	QAQC report
CGAO	2011	MC (6)	NAL	ALS	Y	Y	Y	SRK 2015/ 2006 Snowden
		MCRC (14)	NAL	ALS	Y	Y	Y	SRK 2015/ 2006 Snowden
Terra Gold	2007	TMCD (1)	SGS		Y	Y	Y	GEOCraft 2005
Terra Gold	2005	MD (2)	ALS	ALS	Y	Y	Y	GEOCraft 2005
Hill 50	2001- 2002	MCP (40)	NAL		Y	Y	Y	Snowden 2005/ GEOCraft 2005
		MD (16)	NAL		Y	Y	Y	Snowden 2005/ GEOCraft 2005
Anglo Gold	1999- 2000	MRC	Assay Corp	Amdel	Y		Y	AngloGold internal Standard monitoring
Kilkenny Gold	1997- 1998	DRC (44)	Assay Corp		N	N	Y	Snowden 2005/ GEOCraft 2005
		GRC (17)	Assay Corp		N	N	Y	Snowden 2005/ GEOCraft 2005
		MCE (21)	Assay Corp		N	N	Y	Snowden 2005/ GEOCraft 2005
		MCW (51)	Assay Corp		N	N	Y	Snowden 2005/ GEOCraft 2005
		MD (19)	Assay Corp		N	N	Y	Snowden 2005/ GEOCraft 2005
		SRR (14)	Assay Corp		N	N	Y	Snowden 2005/ GEOCraft 2005

Kalmet	1993- 199	KR (282)	ALS	Assay Corp	N	N	Y	GEOCraft 1996/2005
		MCP (265)	ALS	ANALA BS	N	N	Y	GEOCraft 1996/2005
		MD (33)	ALS	ALS	N	N	Y	GEOCraft 1996/2005
		MRB (216)	ALS		N	N	Y	GEOCraft 1996/2005

Company	Year	Program	Laboratory	Check Lab	CRM analysis checked	Blank analysis checked	Duplicate analysis checked	QAQC report
		MRC (36)	ALS	ALS	N	N	Y	GEOCraft 1996/ 2005
		SWM (9)	ALS		N	N	Y	GEOCraft 1996/ 2005
Placer	1989-1991	WD (20)	Classic Laboratories	ALS	N	N	Y	GEOCraft 1996/ 2005
		WP (40)	Classic Laboratories	ALS	N	N	Y	GEOCraft 1996/ 2005

12.1 Site Visit

Kirsty Sheerin of SRK visited site between the 3rd and 7th August 2015 in order to examine and check the logged core that was still available at site (Table 12-2). Observations are detailed below:

At the immediate footwall to the mineralized zone shale units were regularly observed. Sometime these were logged as such in the primary lithology field, and sometime they were captured in the alteration/structure or comments fields of the database. The shale units are generally very sheared. Where the shale units correlate with mineralization it is possible the low hardness of the shale compared to the sandstone and tuff created a zone of weakness where shearing and subsequent mineralization has occurred. They are generally devoid of veining, so any mineralization is associated with the shale itself, but whether it pre or post-dates the shearing is not known.

From a competency point of view, there appear to be two reasons the footwall is less competent then the tuff. The tuff has generally the same composition/provenance as the sandstone footwall (felsic volcanic), but it is obviously more brecciated in texture. This has allowed the silica and chlorite alteration to penetrate the tuff more. This in turn has increased its hardness. In comparison, the footwall is composed of layers of sandstone, siltstone and shales. These individual rock types are more compacted than the tuff and therefore have been less altered, except between the rock types where the differences in grain size is more pronounced. This has allowed regular shearing along the lithology boundaries within the footwall to occur (particularly where the shale is located) and therefore a less component footwall. This variation between sandstone, siltstone and shale doesn' t appear to have been logged consistently, but it would be of significant use from a mining perspective, particularly within the first 10-15m of the footwall contact.

The isolated mineralization observed outside the main corridor of mineralization was also investigated. While there were only a few instances of this in the available core, generally any mineralization was associated with isolated veining or a shear/fault zone.

Overall the logging contained in the database correlated well with the main lithological contact boundaries observed in the core. There were slight discrepancies in some areas (generally a lack of detail), but none which SRK deemed to be inappropriate for use in a resource model.

Table 12-2: Holes and intervals checked

BHID	From	To	Meters Logged
MC002	284.5	316.47	31.97
MC006	563.4	587.29	23.89
MC004	430.31	463.08	32.77
MD009	96.5	113.8	17.30

BHID	From	To	Meters Logged
MD035	435.72	477.55	41.83
MD038	408	450.3	42.30
MD042	337.25	370.8	33.55
MD027	180.1	218.77	38.67
MC005	578.33	621.41	43.08
MD057	349.9	382.5	32.60
MD051	435.65	473.65	38.00
MD045W1	490.5	520	29.50
MD039	360.8	385.03	24.23
Total			**429.69**

The logging correlated well with what was perceived to be the main lithological contact boundaries. There were slight discrepancies in some areas (generally a lack of detail).

At the immediate footwall to the mineralized zone regular shale units were observed. Occasionally they were logged as such, and sometimes it was captured in the alteration/structure. Generally the shales were quite sheared so this is understandable. Where they correlate with mineralization it is possible the low hardness of the shale compared to the sandstone and tuff created a zone of weakness where shearing and subsequent mineralization has occurred. They are generally devoid of veining, so any mineralization is associated with the shale itself, but whether it pre or post-dates the shearing is not clear.

From a competency perspective, there appear to be two reasons the footwall is less competent then the tuff. The tuff has similar composition/provenance as the sandstone footwall (felsic volcanic), but it is obviously more brecciated in texture. This has allowed the silica and chlorite alteration to penetrate the tuff more pervasively. This has subsequently increased its hardness. In comparison, the footwall is composed of layers of sandstone, siltstone and shales. These individual rock types are more compacted than the tuff and therefore have been less altered, except between the lithologies where the difference in grain size is more pronounced. This has allowed regular shearing along the lithology boundaries within the footwall to occur (particularly where the shale is located) and therefore a less component footwall. This variation between sandstone, siltstone and shale doesn' t appear to have been logged consistently, but it would be of significant use from a mining perspective, particularly within the first 10 – 15 m of the footwall contact.

The isolated mineralization observed outside the wireframed 0.1 g/t halo was also investigated. Only a few instances of this were available at the core shed to inspect, however, the few examples seen suggest that generally any mineralization is associated with isolated veining or a shear/fault zone.

The information gathered from the validation logging was then cross checked thoroughly with the database, and SRK believe the geology model created was appropriate for use as the basis of the resource estimate. Any variations observed with the logging will be used to better understand the genesis of the deposit and help design a future infill drill program.

12.2 2011 Newmarket Gold soil sampling program

Newmarket Gold utilize specialized industry computer software to manage its drillhole and assay database and employ dedicated personnel to manage the database and apply appropriate QAQC procedures to maintain the integrity of the data. Data is assessed for errors against standards and blanks prior to loading into Maxwell GeoServices Datashed™ database software. Data is then spatially assessed in commercially available mining software package Micromine™ for any other questionable results.

Previously, consultants have completed various database checks, which have not identified any reportable errors, which would have raised any concerns about the integrity of the data. During the preparation of this report, which has included search and lookup of assay results, generation of plans and sections and estimation of Mineral Resources, the Qualified Persons did not encounter any difficulties with the database; SRK believes the historical data/database has been verified to a sufficient level to permit its use and confidence in its reliability.

Wherever possible Newmarket Gold has also conducted on ground checks of data, this includes there surveying of historic drill collars and previously mined open pits. The checking of the open pits has involved the use of a surveyor with a depth sounder to test the bottom of the pit against previous pit pickups. This was done to ensure an accurate depletion of the Mineral Resource.

During the past 2 years Newmarket Gold has spent a large amount of time and money reviewing all historic data in both hard and soft copy forms. This has given the Company a much better understanding of the original data that is available for cross checking and review.

12.3 Sampling prior to 2011

Access software was implemented to manage the Maud Creek database in 2006. The software includes a strict, controlled and structured set of fields and columns to manage the data flow, and checks to alert the database manager of any data importation issues.

The geological interpretation, core logging facility and core storage areas were inspected by Snowden in 2006. In all instances the lithologies, mineralization, alteration and sample intervals were found to agree with the drill logs.

Snowden reviewed the database and confirmed that the data extracted for resource estimation matched the primary database records. Overall the review in 2006 concluded that the data has been verified to a sufficient level to permit its use in a CIM compliant resource estimate.

In 2006 Snowden reviewed all previous drilling data and concluded that the lack of documented and relevant QAQC data and protocols was material to the previous estimate, and until addressed would impact upon the ability to classify the resource estimate with greater confidence than Inferred. Based on this advice a resampling program was implemented by GBS, whereby remaining core was resampled and assayed together with the submission of independent certified reference materials (CRMs).

In July 2005 179 previously cut and analyzed core intervals were resampled and submitted to SGS laboratory in Perth for analysis along with a number of CRMs. The results of the program are detailed in the Technical Report Maud Creek Project Drill-Hole Data Validation for Resource Assessment by Andrew Milne of GeoCraft Pty Ltd dated August 2005. Standard fire assay analysis was undertaken on the 179 samples, and then 57 samples were re-assayed using screen fire techniques to compare against the corresponding fire assay analysis.

The following concerns regarding the program were identified:

- Sampling and analysis of different parts of the core,
- Different proportions (half vs a quarter) of the core in some cases; and
- Different Laboratories performing the initial (ALS or Assay Corp) and Resample (SGS) analysis.

Figure 12-1 and Figure 12-2 are precision plots comparing the original data vs the resample and the fire assay data vs the screen fir e data, respectively. Figure 12-1 shows about 30% of the data plotting above the 20% precision line, which given the concerns raised above is an acceptable level.

Figure 12-2 suggests that coarse gold is not affecting the fire assay results and that they can be considered acceptable for use.

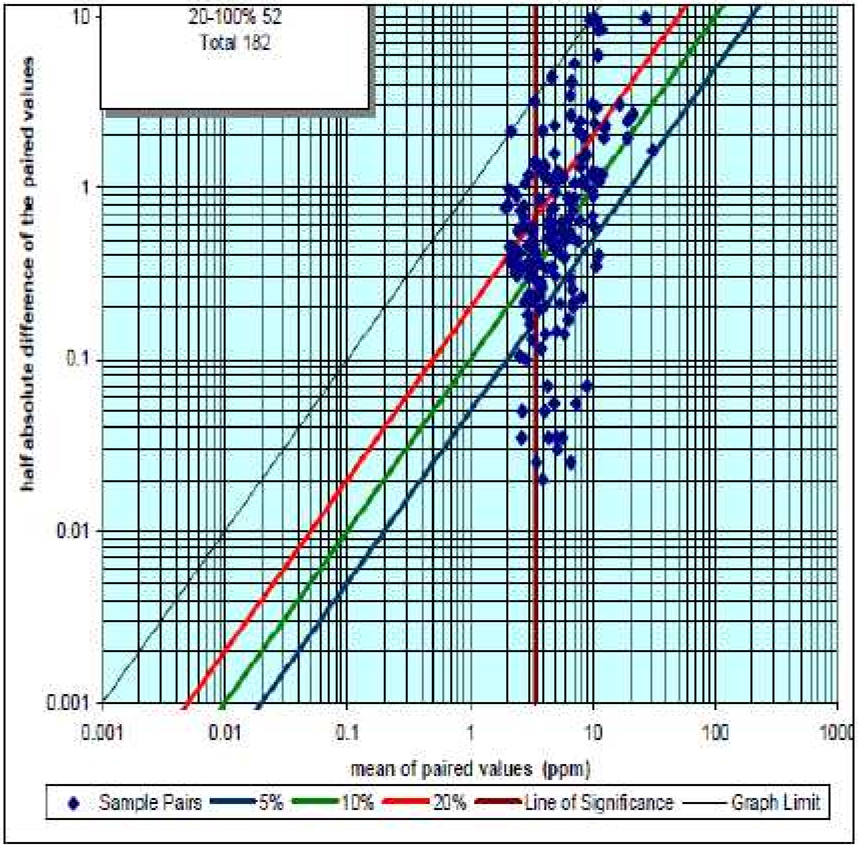


Figure 12-1: Precision plot original vs resample

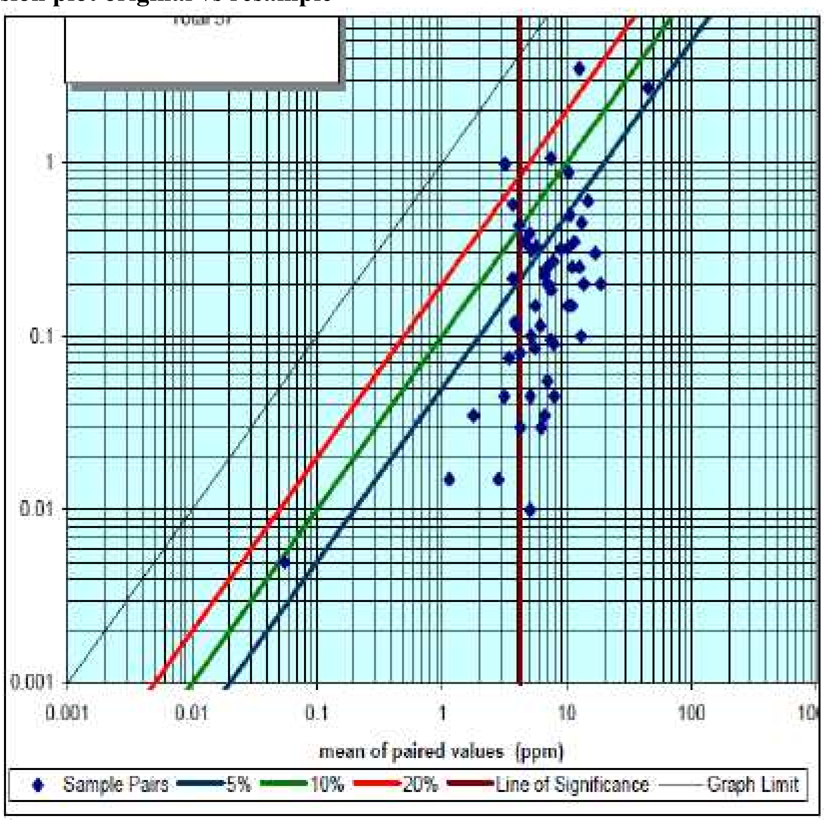


Figure 12-2: Precision plot fire assay vs screen fire assay

13 Mineral Processing and Metallurgical Testing

13.1 Metallurgical Testing

An extensive program of metallurgical testing was carried out from 1994 through to 2006. Much of the focus and testing was on downstream oxidation processes on refractory mineralization, such as BIOX® and the GEOCOAT® Process. This summary of metallurgical testing considers only those parts relevant to the current flowsheet selection, i.e.:

- Crushing and grinding
- Gravity recovery
- Flotation
- Tailings and concentrate dewatering

Direct cyanidation leaching of mineralization and concentrates was tested on the fresh (sulphide) mineralization with poor results and is omitted from this summary.

The major body of work was undertaken for Kalmet Resources N.L. by Ammtec in 1996 – 1998. Nine separate reports focussed on flotation testing, while only one dealt directly with SAG milling.

SAG milling was not tested in detail because the previous owners envisaged a relatively low throughput processing facility (300 ktpa) and using a conventional crushing plant that would suitable for both an oxide mineralization gold heap leach circuit as well as a crushed product size suitable as mill feed. The lack of SAG mill testing is not considered an issue because processing is through the Union Reefs which incorporates three stage of crushing and closed circuit ball milling.

13.1.1 Comminution

Three reports cover measurements of physical parameters for crushing and grinding:

- Ammtec report A5161 "Metallurgical testwork on variability samples VL 5-8 and VF 1- 8 from the Maud Creek Project for Kalmet Resources" (December 1996)
- Ammtec report A6076 "Metallurgical testing of variability samples VF9 – VF15 from the Maud Creek Gold Project for Kilkenny Gold NL" (February 1998)
- Ammtec report A6443, "SAG milling testwork associated with the Maud Creek Gold Project for Kilkenny Gold NL" (October 1998).

Crushing

Reports A5161 and A6443 include measurements of crushing work index (CWi) and unconfined compressive strength (UCS).

Report A5161 describes samples VL 5 – 8, selected from hole MD20 intervals to represent a profile of increasing depth and sulphur level (less oxidised). Ten specimens were selected from these intervals for crushing work index tests and 5 specimens were selected for UCS tests. Results are as presented in Table 13-1.

Table 13-1: Crushing Test results from Report A5161

Sample	Drillhole	Depth (m)	Ore type	CWi (kWh/t)	Lithology
CWi-1	MD21	28.5	Oxide/transition	16.2	Oxidised tuff, quarts & carbonate
CWi-2	MD21	32.7	Transition	8.5	Breccia tuff, quartz carbonate, graphite
CWi-3	MD21	41.5	Transition	11.1	Massive quartz, graphitic, minor pyrite & arsenopyrite
CWi-4	MD20	67.3	Primary	8.2	Graphitic quartz carbonate stockwork, arsenopyrite
CWi-5	MD20	82.0	Primary	10.9	Massive quartz, pyrite, arsenopyrite,
CWi-6	MD14	86.1	Primary	13.5	Tuff, weak stockwork, pyrite
CWi-7	MD14	92.0	Primary	10.0	Tuff
CWi-8	MD14	105.1	Primary	8.5	Massive quartz vein, minor graphite
CWi-9	MD14	109.2	Primary	8.9	Quartz breccia, graphite, pyrite, arsenopyrite
CWi-10	MD14	112.5	Primary	5.9	Graphite quartz stockwork, pyrite, arsenopyrite
			Average	**10.2**	
			Maximum	**16.2**	
			Minimum	**5.9**	
			75th percentile	**11.05**	

The UCS results ranged from 52 – 285 MPa with an average of 163 MPa.

Report A6443 describes the samples tested as from '5 trays of recently drilled HQ core'. Intervals are specified but the exact drillhole was not identified. The core includes oxide, main lode and hanging wall intervals. Four sulphide core specimens were tested for UCS; three from the main lode and one from the hanging wall. Sixteen sulphide core specimens were tested for crushing work index (CWi). Results are as follows, they exclude the oxide sample test results shown in Table 13-2.

Table 13-2: Crushing Test Results from Report A6443 (Oxides Excluded)

	CWi (kWh/t)	UCS (MPa)
Average	10.7	132
Maximum	23.0	182
Minimum	4.5	106
75th percentile	13.2	139

These results indicate moderate average power requirements, but with wide variation. The UCS results would be classified as strong (60 – 200 MPa), with the maximum result approaching the very strong level (>200 MPa) but is not at levels that cause crushing difficulties with appropriate equipment selection. An additional sample VL5-8 had additional UCS tests undertaken on it. The deepest sample demonstrated very high competency (285 MPa) and a 75th percentile of 208 MPa. The combined UCS 75th percentile used for the process design criteria (PDC) is 182 MPa.

The average CWi is only moderate in strength but the maximum level is would be classified as strong. In both cases (UCS and CWi), this suggests more tests would be required to get a reliable average. In the absence of further data, a conservative value has been chosen for design. The crushing testwork results confirm that the Union Reefs three stage crushing circuit is capable of processing the hard Maud Creek fresh mineralization. It may be beneficial to blend the softer Maud Creek oxides with the Cosmo underground ores if there are any material handling or viscosity concerns.

Grinding

Ammtec report A5161 describes composite samples VF4/6 being subjected to measurements of abrasion index and bond rod and ball mill work indices (BRMWi and BBMWi). The composite was a 50/50 blend of primary mineralization sample VF4 and primary footwall sample VF6:

VF 4: Drillhole MD 20, (61.0 - 74.0 m, 80.0 - 83.7 m)

VF 6: Drillhole MD 14, (98.9 - 116.5 m).

These samples had quite high gold grades of 10.83 g/t and 12.31 g/t respectively.

Ammtec report A6076 describes the BBMWi testing of composites VF10 and VF14. The sample origin is provided as:

VF 10: Drillholes MD 40 (323.3 – 329.8 m), MD 41 (314.55 – 319.0 m), MD 44 (325.0 – 337.0 m)

VF 14: Drillholes MD 35 (459.4 – 464.8 m), MD 37 (463.0 – 469.1 m), MD 38 (429.0 – 436.0 m)

Results from the two reports are summarised in Table 13-3.

Table 13-3: Grinding Test Results from Reports A5161 and A6076

	Average depth (m)	BRMWi (kWh/t)	BBMWi (kWh/t)
Earlier work		16.65	17.73
VF4/6	78	17.8	18.1
VF10	283		18.6
VF14	392		19.64
Average		19.0	18.8

Specific grinding power consumption was also recorded in the flotation pilot plant runs:

1996 pilot run: 12.24 kWh/t

1997 pilot run: 11.95 kWh/t

The equipment and feed size were not in accordance with the standard Bond work index methods, so the results should be treated with caution, however they are indicative. In summary, all the comminution test programs indicate a moderately hard, highly abrasive mineralization requiring high grinding energy and highly variable crushing energy.

Summary data from John MacIntyre' s comminution evaluation report are presented in Table 13-4.

Table 13-4: Summary of Comminution Results from J MacIntyre Metallurgical Evaluation (Sept '98)

		Oxide	Primary
Bond crushing work index	kWh/t	5.0	10.2
Bond Rod mill work index	kWh/t	16.65	19.0
Bond Ball mill work index	kWh/t	17.73	18.8
Abrasion index		0.489	0.678
Unconfined compressive strength	MPa	49	163

The following observations are reproduced from the MacIntyre report:

1. *The average Bond crushing work index values for both the oxide and primary zones are low. Although the database is limited the primary zone crushing work index does not appear to increase with depth to a vertical depth of 100 m.*

2. *Both the oxide and primary zone rod and ball mill work index values are high. No rod mill work index value has been measured on samples obtained from a depth greater than 78 m. The primary zone ball mill work index values do not appear to be depth sensitive to depth of 392 m.*

3. *The oxide abrasion index is above average and the primary zone abrasion index is high.*

4. *The unconfined compressive strength (UCS) values are strongly dependent on depth, increasing from 37 MPa at 5 m depth to a very high value of 285 MPa at 90 m depth. No UCS values have been measured on samples obtained from a depth greater than 90 m.*

Subsequent to the MacIntyre report, Ammtec report A6443 described abrasion index (Ai), milling work index and JK drop weight tests on a "comminution composite sample" made up of HQ core from the main lode and hanging wall (oxides excluded). As with the crushing work, the intervals are specified and come from a single, unidentified drillhole. Results were very similar to the earlier reports as shown in Table 13-5.

Table 13-5: Comminution Results from Report A6443

	Ai	BRMWi (kWh/t)	BBMWi (kWh/t)
Maud Creek 'Comminution composite'	0.6487	18.3	19.94

The same composite sample was used for JK Drop Weight tests, giving the following results:

A: 72.8
b: 0.68
Axb: 49.5
ta: 0.41

These parameters define the ore-specific breakage function which can be input to the JK simulation software to predict SAG mill performance and sizings. The tests show this sample to be moderately hard.

The grinding testwork results confirm that the Union Reefs three stage crushing and closed circuit ball milling circuit is capable of processing the hard Maud Creek fresh mineralization. There is a large amount of installed milling power to meet the required grinding power demand. Future tests should be based on geo-metallurgical domains in order to match grinding requirements with the mine plan and to ensure sample representivity for the new mine plan.

Relatively conservative comminution parameters have been incorporated into the design criteria. No significant risks are considered in this aspect of the testwork. There is sufficient comminution capacity at Union Reefs. Additional testwork is not considered to be essential but would provide further confidence and optimisation of the design. Specialist comminution modelling of the Maud Creek mineralization through the Union Reefs circuit is recommended during the next phase of study to confirm the capacity, likely throughput rates and improve confidence in the operating costs.

13.1.2 Gravity Gold Recovery

Gravity Recoverable Gold (GRG) testwork was included in the three flotation pilot plant runs as well as several of the batch testwork programs.

Several preliminary programs were conducted by Amdel and Metcon in 1994. Gravity recovery was found to be significant but highly variable.

Ammtec report A4997 "Optimisation flotation testing of Maud Creek primary gold ore for Kalmet Resources" (May 1996) describes flotation and gravity testing of a composite prepared from drillhole WD 16, 122 – 144m. Batch tests were done in a Knelson (gravity) Concentrator after crushing the sample to 100% passing 1mm, giving 11.36% Au recovery.

Ammtec report A5161 "Metallurgical testwork on variability samples VL 5-8 and VF 1-8 from the Maud Creek Project for Kalmet Resources" (December 1996) included gravity pre-treatment by Knelson Concentrator on samples designated VF 1 – 8. Gravity gold recovery varied from 2.05% to 74.05% on the VF5 sample, noted as being high in carbonates. Summary results are provided in Table 13-6.

The high carbonate sample VF5 was further investigated in Ammtec report A6260 "Flotation optimisation work associated with the Maud Creek gold project for Kilkenny gold NL" (June 1998). As previously demonstrated, very high gravity recovery was observed. A 65.7% recovery of feed gold was recorded from a Knelson concentrator treating P80 500 micron mineralization.

The first pilot plant operation is described in Ammtec report A4952 "Pilot scale flotation testing of Maud Creek primary gold ore for Kalmet Resources" (March 1996). The bulk composite used had a relatively high gold head grade of 9.88 g/t (average). Gravity gold recovery equivalent to 14.5% of the feed was reported. There was also free gold found in the flash flotation concentrate, but this was deemed too fine to be gravity recoverable.

The second pilot plant also included gravity recovery, as described in Ammtec Report A5367 (Part A), "Pilot scale flotation testing of Maud Creek primary ore for Kalmet Resources NL" (March 1997). Gravity recovery equivalent to 14.5% of the feed ore was reported.

Ammtec report A6076 "Metallurgical testing of variability samples VF9 – VF15 from the Maud Creek gold project for Kilkenny Gold NL" (February 1998) included a series of gravity recovery tests. Unlike previous tests, results were generally poor. However, the gravity tails from these tests all responded well to flotation and overall recoveries remained high. A summary table is provided in Table 13-7.

The J MacIntyre report (Sept 98) makes the following observations in respect to gravity gold recovery:

1. *The amount of gravity gold recovered generally increases with gold head grade. A minimum amount of gravity gold may be recovered for gold head grades generally less than 5.6 g/t.*
2. *The first and second pilot plants recovered ...15.8% average....of the total gold.*
3. *The first and second pilot plants recovered 1.430 g/t and 0.975 g/t (1.203 g/t average) of their head grades of 7.837 g/t and 7.373 g/t (7.605- g/t average). That is 18.2% and 13.2% (15.8% average) of the total gold that was recovered as amalgam gold.*
4. *The amalgam gold-gold head grade relationship predicts that approximately 1.55 g/t or 20.4% of the gold is recovered as amalgam gold for the average pilot plant head grade of 7.605 g/t. This relationship has been adjusted downwards by 0.35 g/t such that it reflects the actual average amount of gold recovered by both the pilot plants. The adjusted relationship therefore predicts that 1.049 g/t or 14.1% of the total gold to be recovered as amalgam gold for a 7.40 g/t head grade.*
5. *The amount of amalgam gold recovered appears to decrease with depth, especially when samples grading more than 10 g/t are excluded from the database.*
6. *The amount of amalgam gold recovered appears to be independent of whether the sample is oxide, transition or primary mineralization.*
7. *The amount of amalgam gold recovered appears to be independent of either the sulphur or arsenic head grade.*

Some later testwork also assessed gravity recovery. IMO project No. 1930 "Scoping testwork on Maud Creek gold ore samples for Harmony gold operations Ltd" (June 2003) included gravity separation of a low (1.75 g/t) and high grade (6.14 g/t) composite. The composites were prepared from intervals of main zone (MZ) and eastern shear (ESZ) from a several different drillholes. Gravity recovery to low grade concentrates was reported as 44% from the low grade and 53% from the high grade. These results are probably not realistic due to the high mass pull reporting to the gravity concentrate but support the inclusion of a gravity recovery circuit and potential for reasonable gravity recovery on grades more likely to be fed to the Processing Plant.

The most recent pilot program is covered in Ammtec report A9911 "Pilot flotation on Maud Creek deposit for Terra Gold Mining Ltd" (February 2006). Composites samples were prepared from drillholes MD63 and MD65. Gravity recovery of 18.9% was recorded from a head grade of 1.94 g/t.

In summary, the Maud Creek contains significant but varying amounts of gravity recoverable gold. The relationship presented by John MacIntyre is as follows:

$$g/t\ amalgam\ gold = 0.\ 760\ x\ (gold\ head\ grade) - 4.58$$

Based on the relationship above, the amount of gravity recoverable gold would be minimal (negative) for an average head grade of 4.38 g/t. This is not supported by much of the testwork, there are significant variations in recoveries in the various testwork that has been completed, and the value and relationship given in the John MacIntyre report is considered too conservative.

A gravity circuit would typically be included in a new plant design for a GRG content above 10% and in this case is further supported given the cost of transporting a flotation concentrate for third party processing. Union Reefs already has a gravity gold circuit and therefore the decision on whether to include it in the design is not relevant. In summary, it is recommended that further gravity recovery testwork be performed on mineralization that more closely reflects the current average head grade. It would help better define the payable gold attributed to the gravity concentrate and the flotation concentrate.

Until further assessment is made on the deposit, a gravity recovery of 20% has been used for design purposes based on variability and pilot plant results. This will be updated after further review in the next stage of study. If gravity recovery proves to be lower than this, testwork demonstrates gold is subsequently recovered in flotation, i.e. overall recovery is relatively robust against variation in the gravity gold recovery.

13.1.3 Flotation

Eleven separate testwork reports describe the flotation programs undertaken for Maud Creek. All were carried out by Ammtec Laboratories. The reports can be separated into several phases of work, being; preliminary, variability and optimisation testwork followed by pilot programs. A significant amount of flotation testwork has been undertaken and is considered to support the PEA. Piloting would normally be considered to be at a feasibility level of assessment. This testwork is the key to the Union Reefs flotation plant upgrade.

Preliminary programs

Ammtec Report A4909 "Preliminary assay and flotation testing of a Maud Creek ore composite for Kalmet Resources" (January 1996)

Ammtec Report A4930 "Preliminary metallurgical testwork on Maud Creek primary gold ore composite for Kalmet Resources N.L." (January 1996)

The first of these (A4909) was a simple flotation test on sample from hole (MD 09) with head grade 7.16 g/t Au. Flotation recovery was very high at 95.55%, to a concentrate containing 128.0 g/t Au.

Cyanide leach extraction on the tail gave only 55.1% recovery.

The second program (A4930) was a single set of rougher flotation batch tests to confirm the suitability of the sample for the first pilot program (A4952). The head grade was quite high (11.1, 8.66 g/t Au). In this sample 94.7% of gold was recovered to a concentrate containing 53.9 g/t Au.

The grind size for both of these test programs was 80% passing 75 microns.

Optimisation programs

The next set of tests are considered optimisation batch testwork programs.

- Ammtec report A4997 "Optimisation flotation testing of Maud Creek primary gold ore for Kalmet Resources" (May 1996)
- Ammtec Report A5376 Part B "Maud Creek carbonate depression flotation testwork for Kalmet Resources" (February 1997)
- Ammtec A6260 "Flotation optimisation work associated with the Maud Creek gold project for Kilkenny gold NL" (June 1998)
- Ammtec report A9617 "Flotation testwork on Maud Creek sample TMCD4002 for Terragold Ltd" (May 2005)

The first program aimed to optimise the grind size, circuit configuration, and reagent scheme. The sample was taken from drillhole WD 16, 122 - 144 m, with an average head grade of 3.73 g/t Au. The optimum reagent scheme was reported to be:

- 125 g/t SIBX (collector)
- 40 g/t AP3477 (collector)
- 50 g/t $CuSO_4$ (activator)

However the performance was not sensitive to either the type or dose of reagent over the ranges tested.

Flash flotation caused only a marginal increase in gold recovery. Cleaner flotation was not included in the program. At the time of testing it was not considered to be necessary.

Grind sensitivity size tests showed a gradual drop in gold recovery with increasing grind size to flotation. From this data, a P80 of 75 microns was selected for subsequent testwork. However this is based on a single drillhole and may not be repeatable.

Report A5376 describes an unsuccessful attempt to depress flotation of carbonates using proprietary reagents. Carbonates are mainly an issue if BIOX® processing is used downstream, where acid can be a major cost.

Report A6260 focussed on the effect of downstream BIOX® processing on flotation. Specifically, using acidic water for flotation and using flotation tails for neutralization of BIOX® liquor. There was also some extra flotation testing on the high carbonate sample VF5. High overall recovery was found to be possible from closed cycle rougher/scavenger tests, but with some sacrifice in concentrate grade.

Report A9617 used intervals from drillhole TMCD04002, 192 - 208 m. Batch flotation tests were done on a composite and also on three separate drillhole intervals with varying gold grades. The composite gave very good flotation performance; with a 95.3% gold recovery to a 190 g/t concentrate.

Recovery by grind size showed no improvement from 106 microns down to 75 micron. This is in contrast with the previous results of A4997. Grind size optimisation between 75 and 150 microns.

The process design criteria has selected a size of 75 microns for recovery and concentrate quality purposes and to reflect pilot plant parameters. There may be justification in relaxing this grind size target marginally in future assessments. Union Reefs capacity is sufficient for any of these grind size target options therefore it can be optimized after start-up. This grind size flexibility is an advantage.

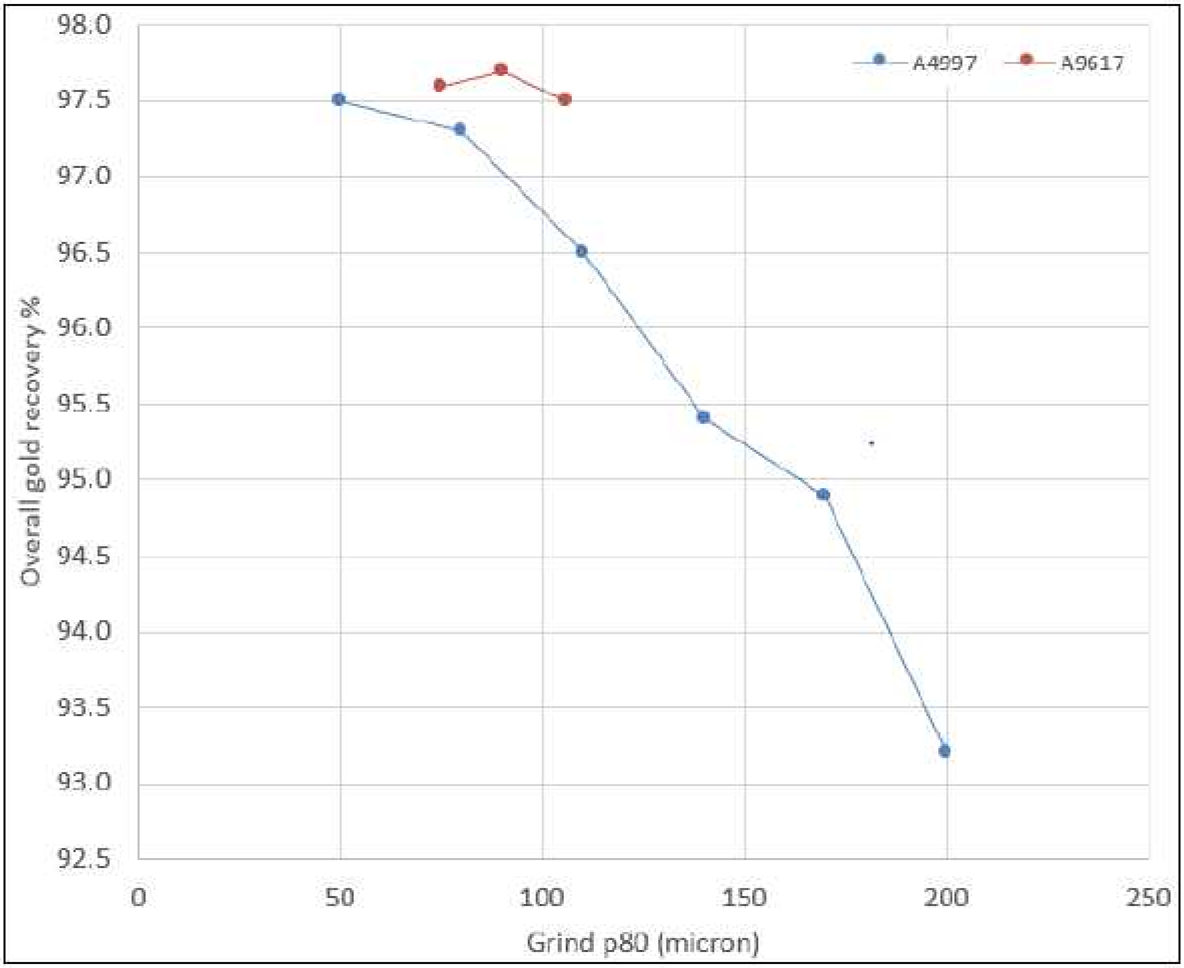


Figure 13-1: Recovery by Grind Size

Source: Extracted from Ammtec Reports A4997 & A9617

Variability programs

- Ammtec report A5161 "Metallurgical testwork on variability samples VL 5-8 and VF 1- 8 from the Maud Creek project for Kalmet Resources" (December 1996).
- Ammtec report A6076 "Metallurgical testing of variability samples VF9 - VF15 from the Maud Creek gold project for Kilkenny Gold NL" (February 1998)

The first of these tested samples taken from varying depth through the mineralization profile, testing first gravity then flotation on the gravity tails. Results are shown below in Table 13-6.

Table 13-6: Flotation Results from Ammtec Report A5161

Sample composite	Ore zone	Drillhole	Average depth (m)	Gold grade (g/t)	Gravity recovery (%)	Flotation recovery (%)	Overall recovery (%)
VF1	Oxide	MD 21	41	15.35	20.91	70.73	91.64
VF2	Oxide/ transition	MD 21	33	7.63	63.75	29.94	93.69
VF3	Trans/ primary	MD 21	33	12.72	41.01	55.64	96.65
VF4	Primary	MD 20	61	10.83	17.24	79.68	96.92
VF5	Primary hanging wall	MD 14	76	7.76	74.05	24.56	98.61
VF6	Primary foot wall	MD 14	93	12.31	46.07	52.06	98.13
VF7	Primary	MD 3	121	16.09	70.73	28.43	99.16
VF8	Primary	MD 27	175	3.36	2.05	93.23	95.28
Average					**41.98**	**54.28**	**96.26**
Maximum					**74.05**	**93.23**	**99.16**
Minimum					**2.05**	**24.56**	**91.64**

Overall recoveries were consistently high, although the gravity /flotation proportion varied widely.

The second variability program, A6076, tested samples VF9-VF15. Flotation recovery on ore was consistently over 95%, with one exception, VF12 at 85.65% . The gravity recovery was significantly lower with only one sample demonstrating any notable GRG. These samples were significantly deeper than the previous optimisation tests.

Table 13-7: Flotation Results from Ammtec Report A6076

Sample composite	Drillhole ID	Average depth (m)	Gold feed grade (g/t)	Gravity recovery (Au%)	Float recovery (Au%)*	Overall recovery (Au%)
VF9	MD 038	298.7	5.94	0.47	95.44	95.46
VF10	MD 040 MD 041 MD 044	326.6 316.8 331.0	5.96	11.99	95.47	96.01
VF11	MD 036 MD 045	290.6 438.5	14.97	0.24	94.67	94.68
VF12	MD 035	383.7	5.89	0.53	85.65	85.73
VF13	MD 034	438.0	4.77	0.38	97.15	97.16
VF14	MD 035 MD 037 MD 038	462.1 466.1 432.5	3.90	1.94	94.85	94.95
VF15	MD 036 MD 045	393.6 474.8	5.14	1.87	96.67	96.73
Average				**2.49**	**94.27**	**94.39**
Maximum				**11.99**	**97.15**	**97.16**
Minimum				**0.24**	**85.65**	**85.73**

*Flotation recovery% from gravity tail

There appears to be a grade versus recovery relationship as shown below in Figure 13-2; however, the correlation co-efficient is poor. Even removing the main outlier does not significantly improve it. At the expected feed grade of approximately 4.38 g/t as per the design criteria, the overall recovery of 95% is considered to be conservative. Further optimisation of the grade recovery relationship (and if possible integration with product grade) under the optimised conditions should be undertaken at the next stage of study.

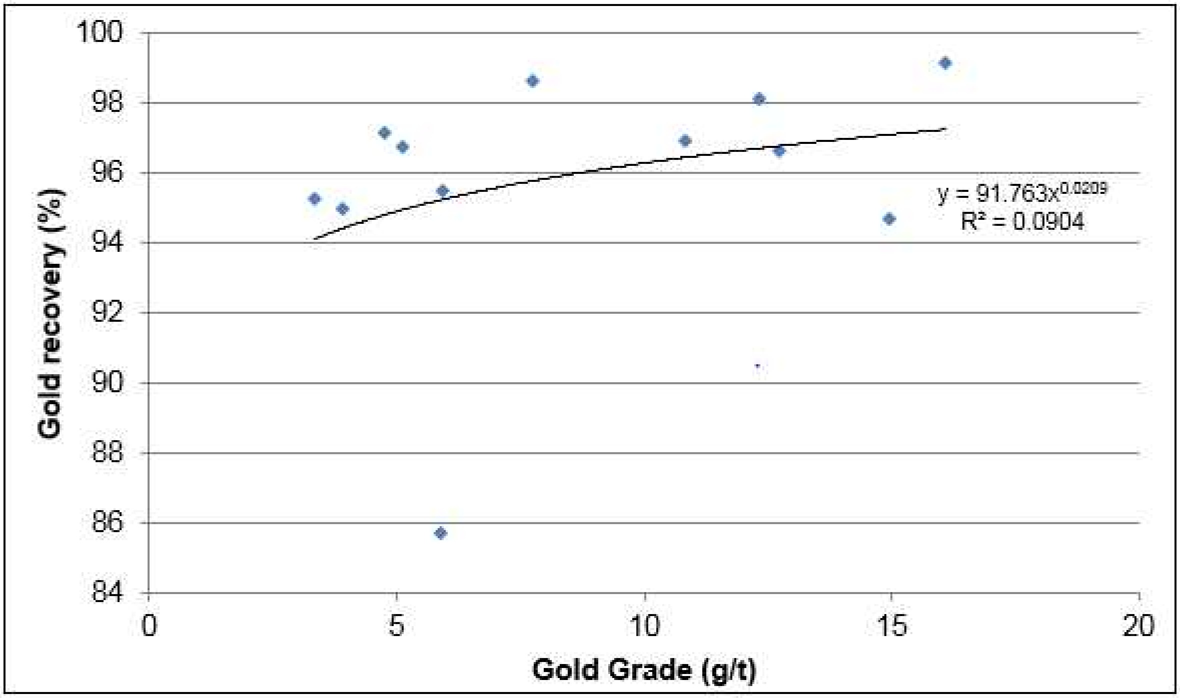


Figure 13-2: Recovery vs. Grade
Source: Ammtec tests A5161 and A6067

Pilot plant programs

Three flotation/gravity pilot plants have been run. In each case, the main purpose was to generate concentrate for BIOX® testing (normally it would be difficult to justify this level of testing for just a conventional concentrator only). The results also provide good process design data.

Ammtec report A4952 “Pilot scale flotation testing of Maud Creek primary gold ore for Kalmet Resources” (March 1996)

Ammtec Report A5367 (Part A) “Pilot scale flotation testing of Maud Creek primary ore for Kalmet Resources NL” (March 1997)

Ammtec A9911 “Pilot flotation on Maud Creek deposit for Terra Gold Mining Ltd” (February 2006)

The first pilot program used a 5 tonne composite sample. The head grade was quite high, with assay measurements of 11.1 and 8.66 g/t Au. Sulphur grades were also quite high at 2.59% .

The pilot testwork circuit included closed circuit grinding with flash flotation and mill discharge passed over a corduroy cloth to collect ‘gravity’ gold. Cyclone overflow passed to rougher, middling and scavenger flotation cells. The reagent scheme consisted of:

50 g/t of copper sulphate added to the mill;

150 g/t of collector SEX stage added as the collector; and

10 g/t of frother MIBC stage added.

The flotation feed P80 was 118 microns, while most concentrate was reground to 17 microns. Gravity recovery was reported as 18.2%, 92.1% from flotation of the gravity tail and 93.5% overall. The flotation concentrate grade contained 51.4 g/t gold and 19.5% sulphur. The overall concentrate gold grade was 63.82 g/t gold.

Flotation gold recovery was 3.5% lower than expected from bench scale tests. This was attributed to the very fine nature of the RC drill chips used.

Flash flotation recovered 41% of the gold, 44% of the sulphur and 38% of the arsenic into 4.0% of the mass.

The second pilot run in 1997 used a bulk sample of lower grade material taken from RC chips, with a lower average Au grade of 6.69 g/t and 1.85% sulphur. Flotation feed P80 ranged from 87 to 100 microns. Flotation recovery varied from 87.3% to 94.8%, with higher recovery corresponding to lower concentrate grade. This supports the future development of the feed and product grade versus recovery relationship once a more extensive data set is available. Combined results from the four survey points are shown in Figure 13-3.

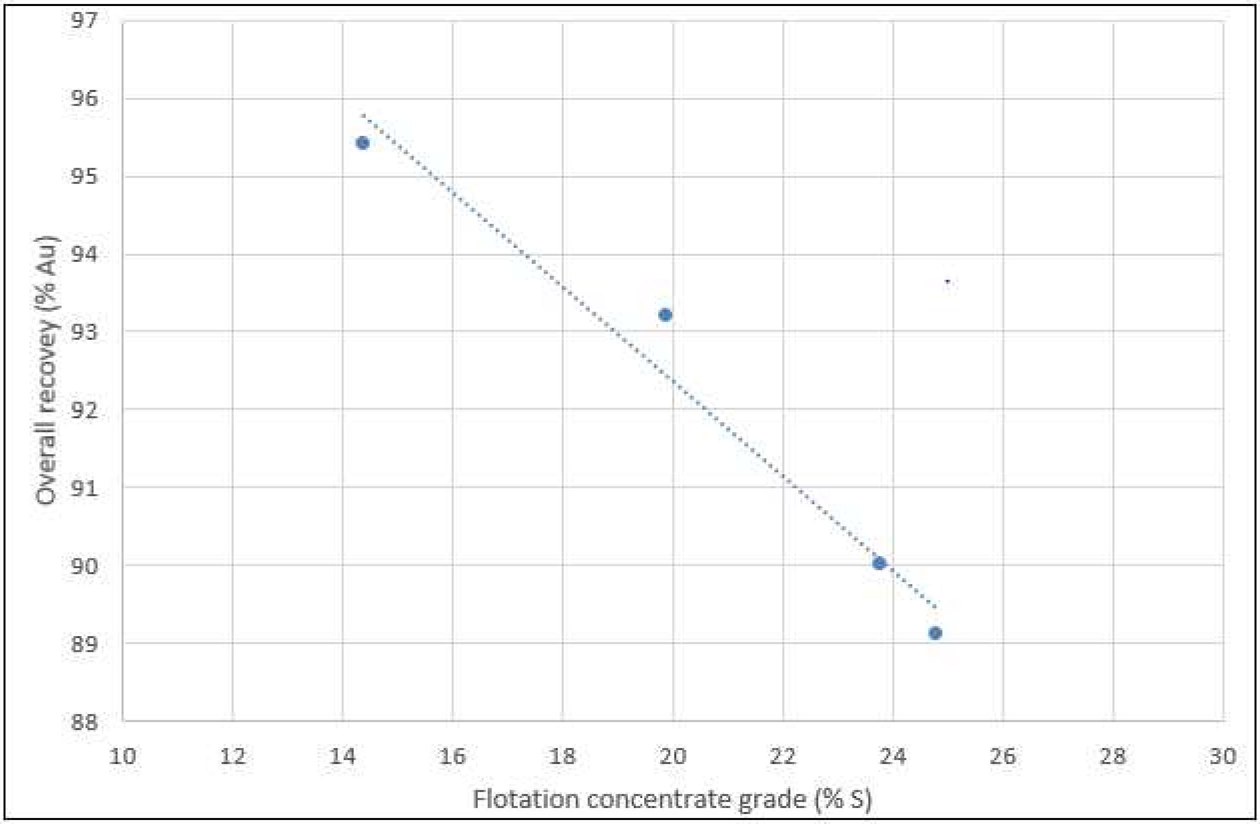


Figure 13-3: 1997 Pilot Plant Survey Results

The lower two recovery points included cleaner flotation while the higher recovery points did not. The cleaner stage was added to increase sulphur grade to over 18 - 20%. It also raised the gold grade from 47.3 g/t to 69.4 g/t. The following points regarding the second pilot plant are extracted from the J MacIntyre evaluation report:

The pilot plant consisted of a gravity concentration stage, a flash flotation stage and a secondary flotation stage. Total flotation residence time was 46 minutes;

Mill feed was crushed to a P100 of 4.0 mm and also had a very fine P80 crushed product size of 1.2 mm;

A bulk secondary float was employed for the first two days of the pilot plant. A cleaning stage was used on the middling and scavenger concentrates for the last two days; and

The plus 180- micron fraction of the flash flotation concentrate (approximately 8.7- kg wet) was the only product reground to a target P100 of 180 microns. The following points are concluded.

1 *An excellent reconciliation exists between the calculated head grade and the drillhole head grade. The calculated head grade of 7.37 g/t agrees within 1% of the drillhole grade of 7.41 g/t.*

2 *A total gravity plus flotation recovery of 91.9% was realised. The first two days recovery of 94.3% was achieved without any cleaning stages – this produced a saleable concentrate grade. This reduced to 89.6% for the last two days when a cleaning stage was employed on the middling and scavenger concentrates.*

3 *13.2% of the total gold was amalgamated gravity concentrate gold.*

4 *93.0% of the gravity tail gold was recovered by flotation in the first two days. This is 3.2% less than the 96.2% gravity tail recovery predicted from Section 2.9.3's grind- recovery relationship. The very fine nature of the RC drill chip would have also contributed to the much lower than predicted recovery.*

5 *Flash flotation recovered 45% of the gold (41% for the first pilot plant), 50% (44%) of the sulphur and 41% (38%) of the arsenic into 3.8% (4.0%) of the mass.*

6 *The sulphur recovery for the first two days of 99.0% is similar to the predicted sulphur recovery of 98.5%. Sulphur recovery reduces to 95.6% when a cleaning stage was employed for the last two days.*

7 *The arsenic recovery of 90.1% for the first two days is similar the 89.4% predicted from Section 2.9.3's grind-recovery relationships. Arsenic recovery reduces to 85.7% when a cleaning stage was employed for the last two days.*

8 *The second pilot plant sample contains 205 ppm of copper, of which 84% is recovered into the combined flotation concentrate at a mean grade 1,801 ppm.*

9 *Days 3 and 4's cleaned concentrate contains 4.4% of the carbonate being 54% of the amount of carbonate contained in Day 1 and 2's un- cleaned concentrate of 8.2%.The cleaned concentrate mass of 7.0% is also 57% of the value of the uncleaned concentrate mass of 12.3% . That is both the uncleaned and cleaned concentrate grades are similar at 6.0% and 5.4% respectively.*

10 *The amount of calcium contained in the concentrate is similar to the carbonate content . That is 8.0% calcium versus 8.2% carbonate for the un-cleaned concentrate and 3.6% calcium versus 4.4% carbonate for the cleaned concentrate.*

11 *The second pilot plant sample contains a negligible amount of mercury. The highest value recorded in the concentrate was 0.038 ppm.*

12 *The combined flotation concentrate's mean P80 of 66 microns is much coarser than the 17 microns for the first pilot plant, but is consistent with the BIOX® requirements.*

The comments regarding carbonates and BIOX® requirements can be ignored if concentrate is to be exported as per the base case. The most recent test program was conducted in 2006 as described by Ammtec report A9911 "Pilot flotation on Maud Creek deposit for Terra Gold Mining Ltd" (February 2006). Tests were conducted on historical drill core from holes MD063 and MD065, drilled in the late 1990s. The condition of the core may have deteriorated to some extent (oxidised) post drilling. The samples had not been refrigerated.

A series of bench scale tests showed high flotation rougher recovery (~93 - 94%) and effective cleaner flotation. The pilot flotation run, however, showed lower recovery:

Gravity gold: 18.9%

Flotation concentrate: 64.6%

Overall recovery: 85.5%

The grade of the final concentrate was reported as 22.2% sulphur and 40.7 g/t gold. It is not clear why the pilot run gave much poorer results than the batch tests, but the results may have been compromised by the age of the sample or the pilot operation/ stability.

13.1.4 Tailings and concentrate dewatering

Thickening tests were carried out by Supaflo (now Outotec) and BPR in support of the BIOX® process design. The more conservative results from the range below are adopted for the design:

Flotation tailings

Flocculant dose: 25 - 35g/t (M-358)

Capacity: 0.7 – 0.8 $t/m^2/h$

Underflow density: 68 – 70%

Flotation concentrate

Flocculant dose: 25- 35g/t (M- E10)

Capacity: 0.6 – 0.7 $t/m^2/h$

Underflow density: 56 – 58%

No data is available on concentrate filtration.

13.2 Future Testwork

The Maud Creek mineralization has already been subject to extensive metallurgical testing, adequate to support this PEA. However there remain some gaps that should be covered to reduce the design risk for future stages if it was decided to undertake additional testwork.

Many tests do not specify the lithological domain of the samples, and some do not specify the sample origin at all. In some cases it could be better to design around the uncertainty rather than undertake further testing. For example, the crusher could be sized for the worst-case hardness rather than attempting to determine an accurate average value.

Much of the comminution testwork was undertaken at shallower depths. It may be worth considering additional comminution testwork on deeper samples if new sample becomes available however the Union Reefs crushing and grinding circuits are adequate to meet the target throughput of the Maud Creek mineralization. At this level of study materials handling testwork is not considered to be essential.

The optimum grind size was determined for conventional crushing followed by ball milling. There may be a considerable saving in the operating cost by choosing a coarser size. This should ideally be conducted on ore-domain composites and supported by economic trade-off analysis. Some specialist comminution modelling using the Union Reefs circuit is recommended for the next stage of design.

The flotation and gravity circuits can be designed from the available results. However, confirmation of gold recovery by mineralization domain would be useful for production forecasting and economic analysis. Furthermore, some flexibility should be built in to the layout to allow for circuit changes, such as retro-fitting flash flotation and/or cleaner flotation. Operating experience and changes in the market conditions/payment terms may justify these measures to increase product grade. Further development of a feed and product grade versus recovery relationship will improve confidence in predicting the overall recovery. This should be undertaken at the next level of study.

A flash flotation circuit has not been incorporated into the design at this level of study. Limited flash testing was done over the life of the project however when tested, such as in the final piloting program, it has been shown to be relatively effective. The flowsheet has been shown to be simple and robust with good product grades without it and to keep the flowsheet simple and reduce costs; it has been excluded at this point. This aspect of the flowsheet will be considered in more detail in the next level of study as it remains a potential opportunity.

Filtration of concentrate samples should be included in future programs. These can be easily added to the flotation confirmation tests and will be valuable for sizing and selection of the filters. It is not considered a risk for design. Conservative assumptions have been made at this time.

Potential customers will most likely want to test samples of concentrate before agreeing to sales terms. Future tests should be used to generate concentrates for marketing purposes.

This study focuses mainly on metallurgy and processing of the sulphide mineralization. For the oxide and transition material (limited tonnage of transitional included in the LoM), controlled potential sulphidation (CPS) should be considered to enable flotation. CPS is an established technology currently used on gold and combined gold/ copper ores at several mines in Australia and overseas. After comminution, mineralization is treated with sodium hydrosulphide (NaHS) which reacts with the oxide minerals, rendering the surface hydrophobic and thus amenable to flotation. This could considerably increase the tonnage treatable through the flotation plant. Reagent consumption rates and recoveries would need to be established by testing.

It is noted that gravity and flotation recovery testwork on transitional mineralization is limited and the metallurgical behaviour is not particularly well understood. It only makes up approximately 230 kt of the overall LoM feed and therefore extensive testing cannot be justified, including sulphidation, but it will make up a large part of the first year' s tonnage so having a reasonable understanding of its performance, particularly the recovery is important for the Project' s cash flow. This needs further consideration at the next level of study. Discounted transitional mineralization recovery of 85% has been used for modelling purposes but there is likely to be a high degree of variability in recovery depending on the level of oxidation. There may be potential to take a flexible approach to processing the transitional mineralization through either the cyanide leach circuits at Union Reefs or even the processing through flotation with the tail processed through the cyanidation circuit if recoveries are poor.

The original focus of the Maud Creek study was on the flotation recovery of the fresh sulphide mineralization. More recently, the opportunity of processing the oxide and some of the transitional mineralization through the Union Reefs cyanidation circuit has presented itself. Additional work is required to confirm the oxide and oxide/transitional recovery through gravity and cyanidation. It only makes up a relatively small proportion of the overall feed but requires more attention at the next phase of study.

13.2.1 Delineation of Mineralization Oxidation Extent

The deposit can be classified three ways; fresh, transitional and oxide mineralization. Fresh mineralization is the primary focus of the study as it makes up the bulk of the LoM tonnage. Small amounts of transitional mineralization will be processed with the fresh mineralization, however most will report to the oxide blend. The total oxide tonnage is relatively low as the bulk of it has been previously mined. It does not justify a standalone CIL/ CIP gold plant but with the Maud Creek base case now being to process through the Union Reefs Processing Plant, the oxide mineralization can now be processed through the existing cyanide leaching facility, providing early cash flow for the Project.

Based on the classifications in Table 13-8, Zones 3 to 5 are to be treated by flotation and will drive the open pit and underground mines. Zones 1 and 2 are expected to be processed through the cyanide leach circuit but any oxide/ transitional mineralization showing poor leach recovery may be processed through the flotation circuit or through flotation and the flotation tail through cyanidation.

Table 13-8: Classification of Zones in Maud Creek Deposit assumptions

Weathering Zone	Description	Sulphur Total (%)	Metallurgical recovery (%)
1	Oxide	0.1	>90%
2	Oxide/transition	0.43	>90%
3	Transition	0.6	85%
4	Transition/fresh	1.26	95%
5	Fresh	1.24	95%

Figure 13-4 below shows the deposit and is divided into the five mineralization types, with light blue indicating Zone 1 oxide mineralization, light green indicating Zone 2 oxide/ transitional mineralization, green indicating Zone 3 transitional mineralization, yellow indicating Zone 4 transitional/fresh mineralization, and red indicating Zone 5 fresh mineralization which makes up the bulk of the overall Project tonnes. Figure 13-4 shows approximately 50m of Zones 1 to 4 mineralization covering the Zone 5 fresh mineralization.

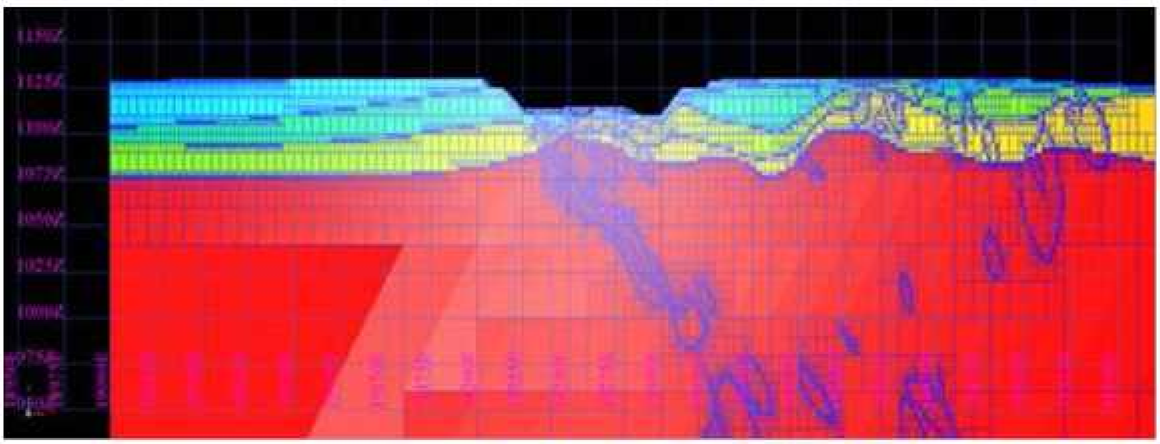


Figure 13-4: Cross section of Maud Creek Deposit

The oxides and oxides/ transitional mineralization in Zones 1 and 2 will be stockpiled separately and processed through the existing Union Reefs oxide circuit. Zones 3 and 4 are processed but have a lower transitional mineralization metallurgical recovery assigned to them.

13.3 Geometallurgy

In order to assess the impact of the Maud Creek mineralogy on reagent consumption and other geometallurgical considerations during processing, a collation of all existing metallurgical testwork was conducted (Table 13-9). Any reports which detailed the drillhole ID and from/to depth of the samples tested were incorporated into a copy of the Maud Creek drillhole assay table. To ensure there was no confusion with the existing multi-element data, any additional elemental data from metallurgical testing was given the prefix 'Met_' . All metallurgically tested intervals, including elemental and processing testwork, were assigned an identifying 'Met Sample ID' ; a combination of the year of the testwork and the 'To' depth of the sample. Where composited samples had been collected, the same value was applied for all intervals, as designated by previous assay sampling.

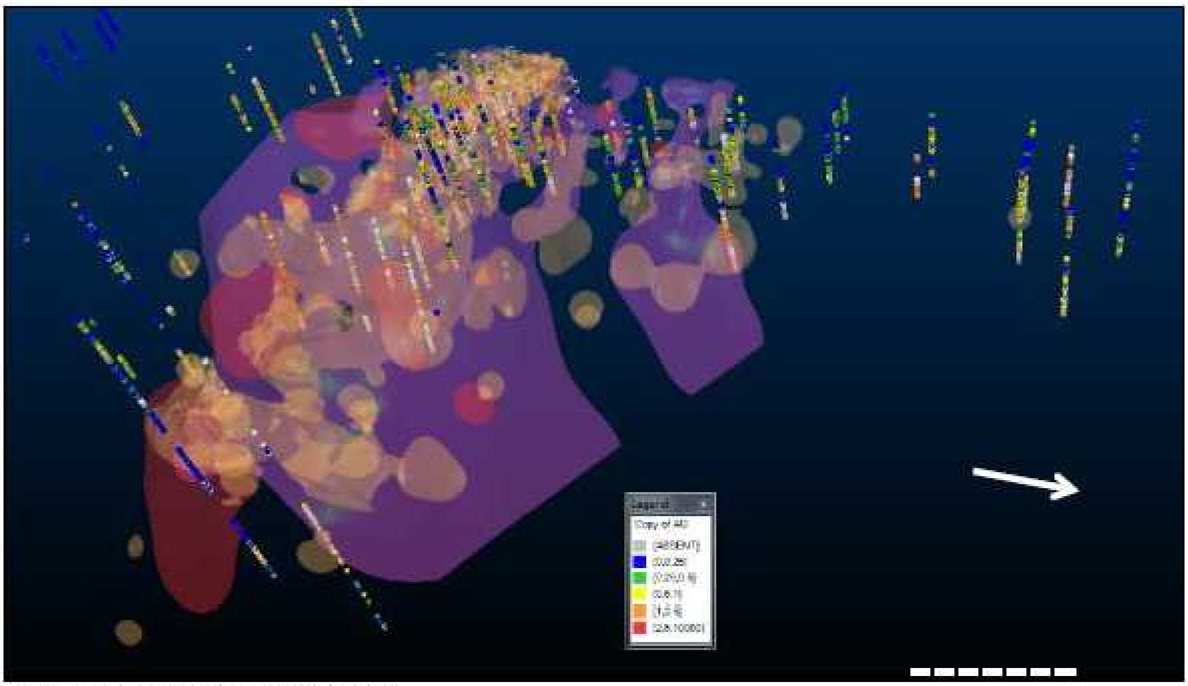
A total of 7,696 values were recorded from the metallurgical reports to create a geometallurgical table. These were then added to the existing 107,677 multi-element values in the assay table. The variables in the geometallurgical database identified as of interest to processing by Simulus include silver, arsenic, bismuth, carbon, carbonate carbon, CO3, organic carbon, total carbon, sulphide sulphur, sulphate sulphur, total sulphur and antimony. Along with collar and survey information this geometallurgical data was then imported into Leapfrog and interpolations creating using the existing structural trends created for the geology model. These interpolations were then imported into Datamine and in conjunction with the drillhole database and lithology logging, amended in cross section to ensure geological considerations were taken into account.

At this point, due to limited data the carbon (carbonate carbon $CO3^{-2}$, organic carbon and total carbon) were combined into one carbon wireframe interpretation, and the sulphur sulphide sulphur, sulphate sulphur and total sulphur combined into one sulphur wireframe interpretation. This exercise was conducted to determine whether there was enough data to create geostatistically robust preliminary geometallurgical domains. Unfortunately, the lack of data, presence of composite values (same value across a large interval) and preference for metallurgical testing to the south and west of the main deposit meant this was not possible.

Arsenic and silver were the only two variables with sufficient samples spread across the deposit, due to multi-element testing during previous drill programs. Arsenic was estimated in the block model using its own variography and kriging parameters, but limited to the mineralized domains previously created based on the gold samples. Due to the low numbers of silver, bismuth, carbon, sulphur and antimony data these variables were was estimated into the same gold bearing domains, rather than into their own domains. Also, due to insufficient data to generate variograms, parameters from the gold variography and kriging neighbourhood were used instead.

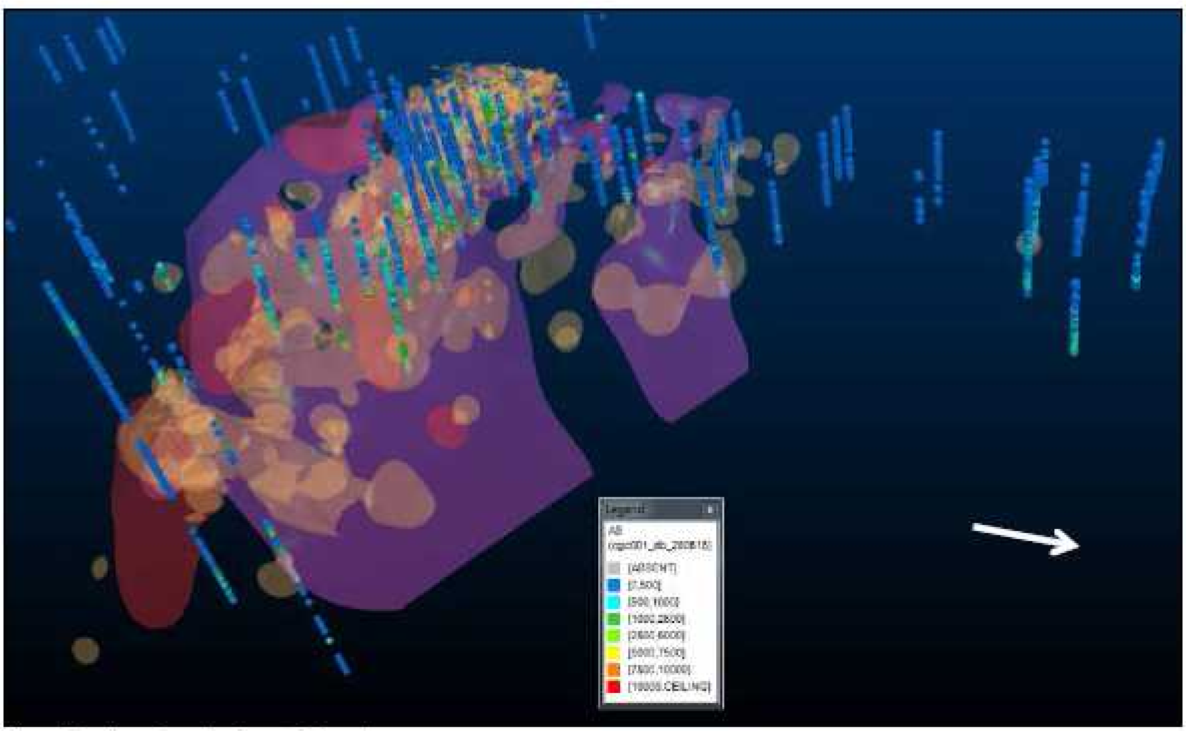
This process allows the model to indicate that, for example, sulphur testing has been conducted in a certain area, but not a quantification of the amount of sulphur present. Figure 13-5 to Figure 13-10 show the distribution of the testing of these variables compared to the main vein, minimum vein and 0.75 g/t Au halo wireframes which were used as mineralized domains for estimation. The estimation of these variables will allow a more targeted approach to the next phase of metallurgical testing, and give an indication of how the geometallurgical variables correlate to the existing lithology and alteration spatially.

For Figure 13-5 to Figure 13-10 below the pink wireframe is the main vein, red wireframe the minimum vein and the orange wireframe the 0.75g/t halo wireframe. All views are long section facing south-east.



View Facing South-East Showing

Figure 13-5: Long section showing Silver Data Compared to Mineralized Domains



View Facing South-East Showing

Figure 13-6: Long section showing Arsenic Data Compared to Mineralized Domains

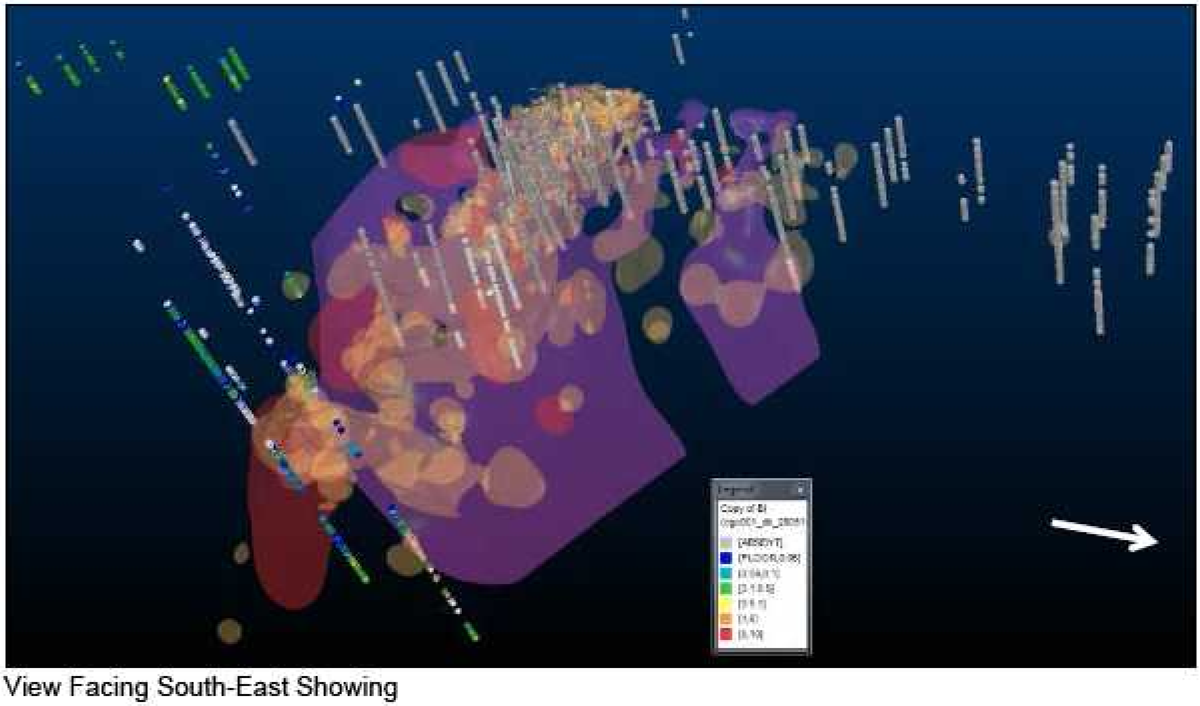


Figure 13-7: Long Section showing Bismuth Data Compared to Mineralized Domains

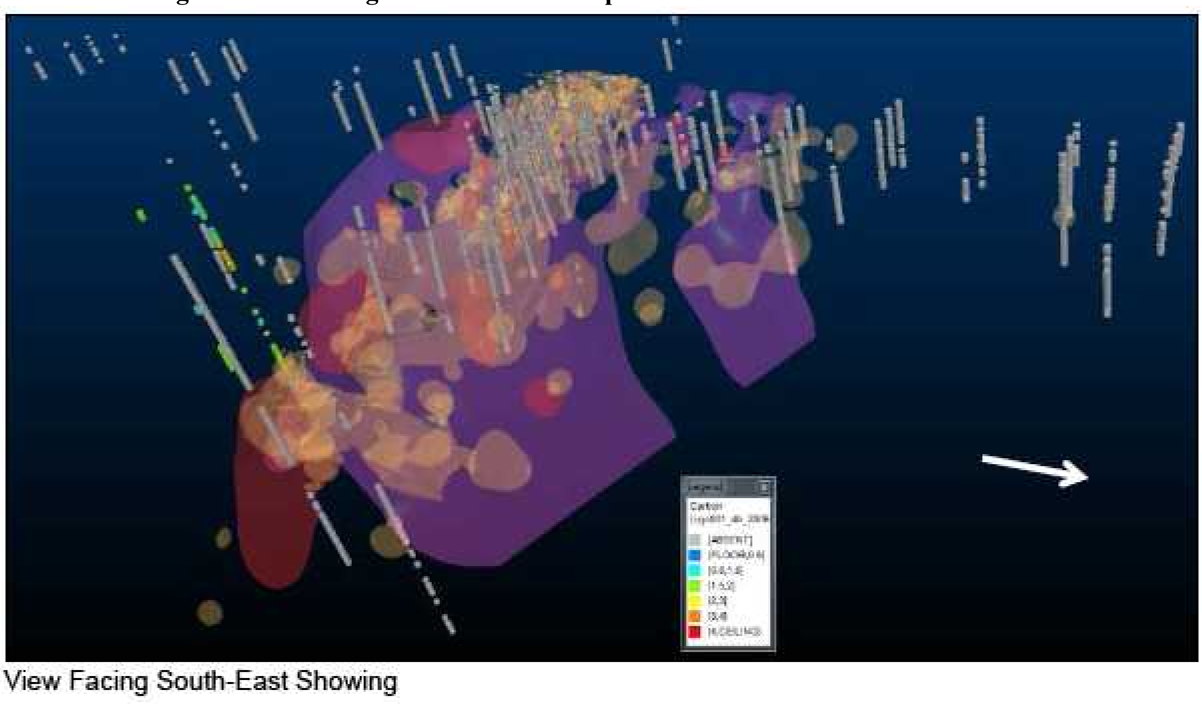
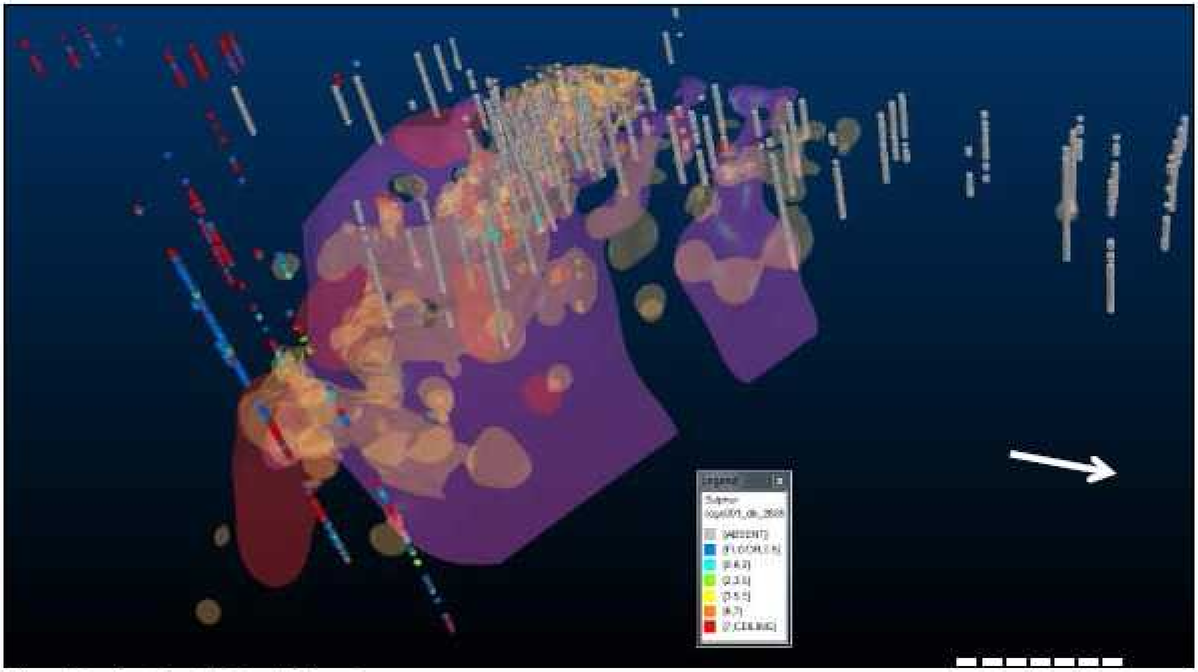
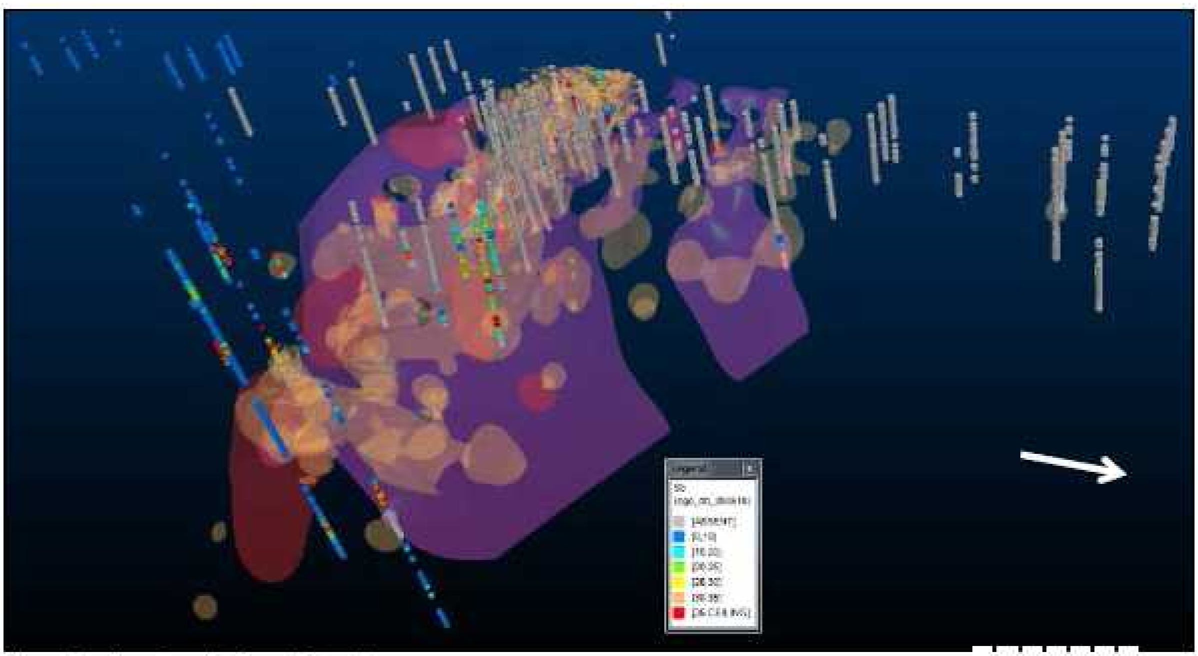


Figure 13-8: Long Section showing Carbon, Carbonate Carbon, CO3, Organic Carbon1 and Total Carbon Data Compared to Mineralized Domains



View Facing South-East Showing

Figure 13-9: Long Section showing Sulphide Sulphur, Sulphate Sulphur and Total Sulphur Data Compared to Mineralized Domains



View Facing South-East Showing

Figure 13-10: Long Section showing Antimony Data Compared to Mineralized Domains

Table 13-9: Historical Metallurgical Reports Used to Construct Geometallurgical Database

Year	Report Name	Company	Number of Samples Tested	Composites	Element Testing	Metallurgical Testing
2005	Maud Creek Flotation Testwork on Maud Creek Sample TMCD04002	Terra Gold	16	16	Au As S Fe	Cyanide recovery
2003	Maud Creek Scoping Testwork on Maud Creek Gold Ore Samples	Harmony Gold	262	14	Au Ag As Cu Pb Zn Hg S Bi Fe Organic C Total C	Cyanide recovery Preg robbing Gravity
1998	Metallurgical Evaluation of the Maud Creek Project	Kilkenny Gold	125	125	As S Fe Organic C CO3	Cyanide recovery Inferred pyrite
1998	Metallurgical Testing of Variability Sample VF9	Kilkenny Gold	89	89		Cyanide recovery
1998	Kilkenny Gold Resources Leach testing - hidden in report #68_116459 Maud Creek, NT, Assays, Lab files, 1996-2005	Kilkenny Gold	21	21		Cyanide recovery Au head and tails
1996	Interim Working Report on the Metallurgical Evaluation of the Maud Creek Gold Project	Kilkenny Gold	44	23	Au Ag As Cu Fe	Bottle roll test Product size Cyanide recovery Cyanide

					Organic C Carbonate Total C Sulphate S Sulphide S Total S	soluble head and residue grade Reagent data Lime consumption Leach kinetics
1996	Maud Creek-Metallurgy Reports	Kalmet Resources	12	12	Au Ag As Cu Fe	

Year	Report Name	Company	Number of Samples Tested	Composites	Element Testing	Metallurgical Testing
					Organic C Total C Sulphate S Sulphide S Total S	
1995	Maud Creek-Metallurgy Reports	Kalmet Resources	10	10	As S Fe	
1994	Maud Creek Metallurgy AMDEL 2994_OCR	Kalmet Resources	99	99	Au Ag As Cu Sb Sulphide S Total S	Gravity Cyanide recovery
1994	Stage 1 Metallurgical Testing Maude Creek	Kalmet Resources	25	25	Au Ag As Cu Sb Pt Pd R Rh Os Ir Organic C Carbonate C	Gravity Inferred pyrite Cyanide recovery

14 Mineral Resource Estimate

14.1 Introduction

The elements to be estimated are Gold (Au), Arsenic (As), Carbon (C), Copper (Cu), Sulphur (S) and Antimony (Sb). The estimation of the elements has been based on assays sourced from drilling data and metallurgical tests where available. The data available as at March 2016 consisted of reverse circulation (RC), diamond core (DD) and rotary air blast (RAB). RAB samples were excluded from the estimation at the exception of 2 samples.

14.2 Lithology and Structural Model

The Maud Creek lithological model was constructed on the local grid coordinates covering dimensions 1,185,000 m (east) and 1,600,000 m (north). The model incorporates several datasets including Diamond and RAB drilling, AngloGold pit mapping and historic SRK aeromagnetic interpretations (SRK, 1998). All datasets were imported into Leapfrog for subsequent 3D modelling. A topography surface was constructed from collar points and used to constrain the top of the lithological model.

The Maud Creek deposit is hosted within the Proterozoic El Sherana Group units and the mineralization hosted at the faulted contact between the Dorothy Volcanic Member and sediments of the Tollis Formation. The Dorothy Volcanic Member strikes approximately north-south and consists of volcanic tuff with minor interbedded zones of sediments. The Tollis Formation strikes north-south and consists of sandstone and metasediments. The deposit is bound to the east by the Maud Creek Dolerite which intrudes the Tuff sequence. A small Andesite body is also observed to the north of the Maud Creek Open Pit. It is located at the faulted contact between the Sandstone and Tuff, forming a discrete body (Figure 14-1, cover units not included). These key units are also overlain by a thin layer of sedimentary cover and Cambrian Volcanics.

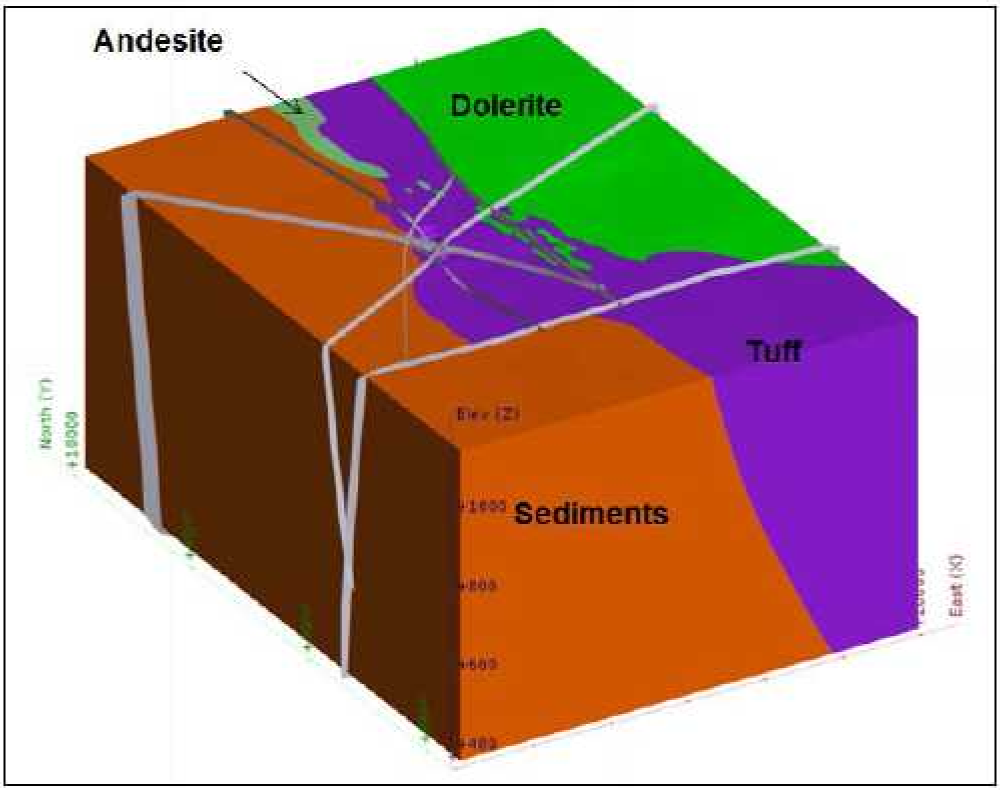


Figure 14-1: Maud Creek Deposit lithology model showing primary units, cover not shown

The key unit formations described above were determined based on the logging code 'Lith Code 1' from the dataset provided by Newmarket Gold. For the purpose of 3D modelling the combination of several lithologies was sometimes required to form a key lithology group (Table 14-1).

Table 14-1: Maud Creek Lithology Model Groupings

Lithology Group	Codes
Cover	ALUV, CLA, CLAY, CLY, GO, LOM, MUD, SOIL, SPLT, SND, CALC, CLCR, LMST, BLT
Dolerite	DLT, DOL, INTD
Tuff	IGM,MAFT,TUF
Sediments (Sandstone)	MSED, GYWK, QTZ, Slst, MDST, SDST, Ssl
Andesite	ANT
Vein	BX, VEBX, VEIN, SHLE

Within the model area eight faults have been identified (Figure 14-2) based on historic AngloGold pit mapping as well as aeromagnetic interpretations conducted by SRK Consulting (SRK, 1998). Orientations of these structures were extracted from the mapped pit data and interpreted based on available datasets. Generally, the faults exhibit reverse movement, with limited offsets in the range of meters. Figure 14-3 shows the interpreted fault architecture, indicating apparent reverse movement along faults. To the south of the Maud Creek Deposit a major east-west structure with sinistral strike-slip movement has been interpreted based on the drilling data and aeromagnetic interpretations. Additional faults are likely present in the modelled area, however only faults which have significant structural control on the deposit have been constructed.

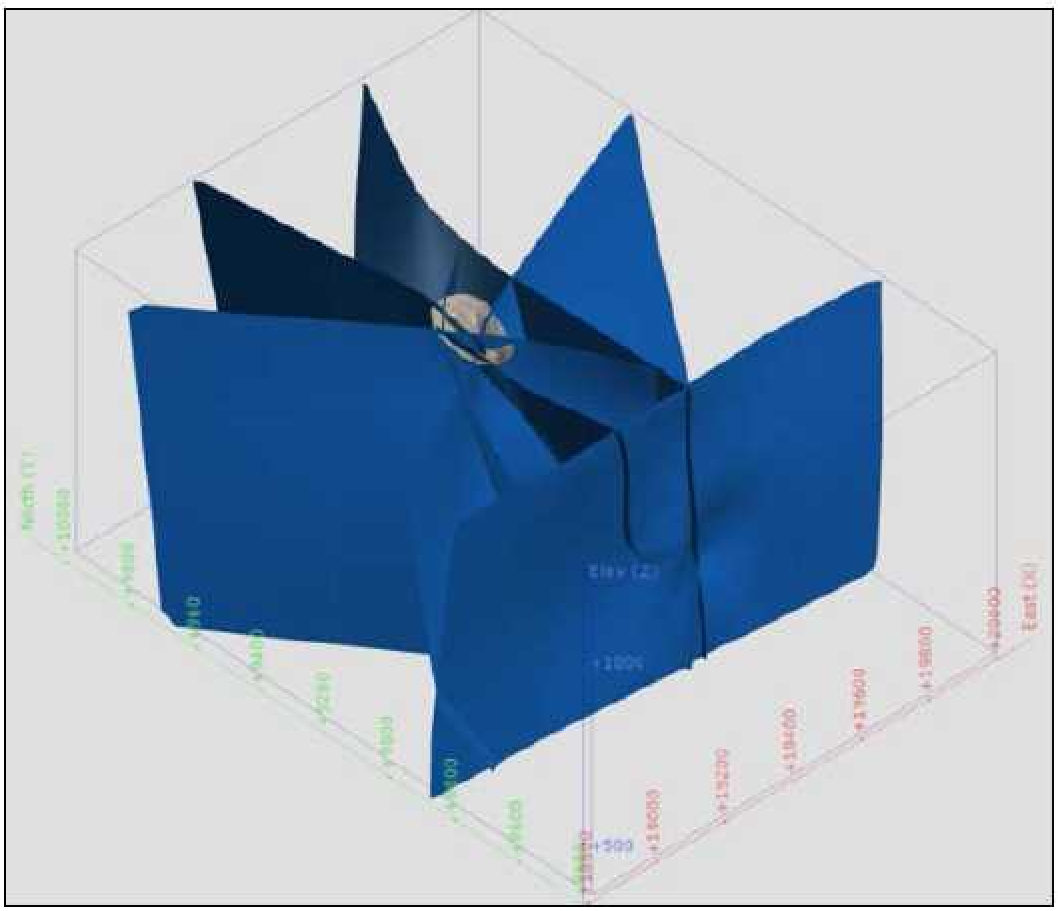


Figure 14-2: Interpreted fault architecture of the Maud Creek Deposit

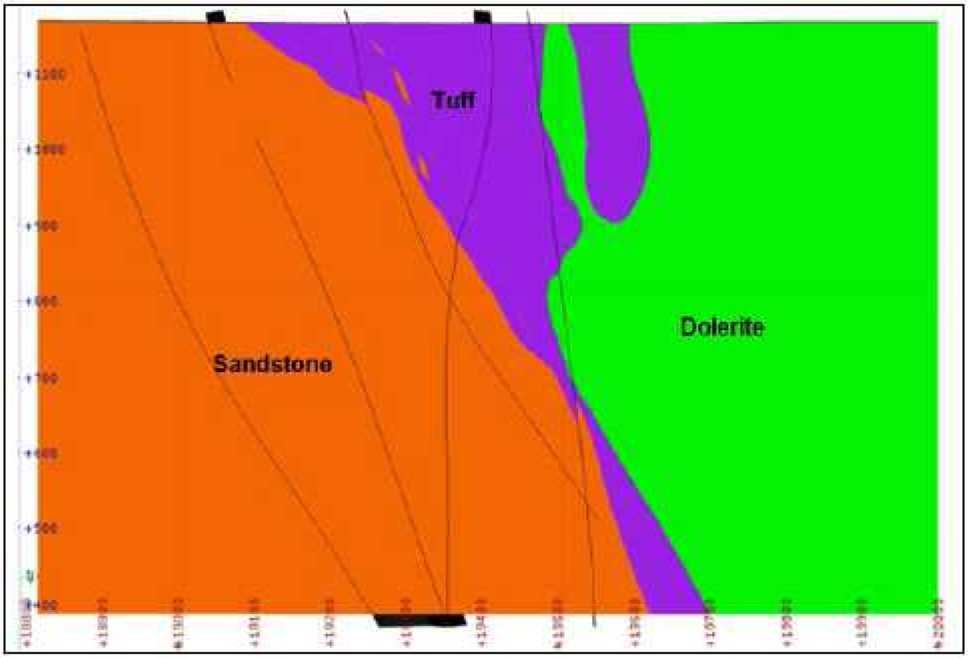


Figure 14-3: Cross section view looking north (northing 9008mN)

14.3 Vein Model

Three veins have been interpreted based on the main lithology logging (Lith Code 1); a continuous primary vein, discrete upper vein and discrete lower vein (Figure 14-4). The veins generally follow the main faulted contact between the Sandstone and Tuff units and have an apparent plunge to the south-east, which follows the contact fault (Figure 14-5). The primary vein typically hosts the highest gold grades and lies within the Sandstone/Tuff contact, however deviations from this contact are evident. The upper and lower veins generally hosts lower gold grades and exhibit limited continuity located above and below the faulted contact. Additional vein material was evident in the logging however these veins typically have distinctly limited continuity and have not been included within the final model. The presence of contact veining decreases to the north of the Anglo Pit, based on available drilling.

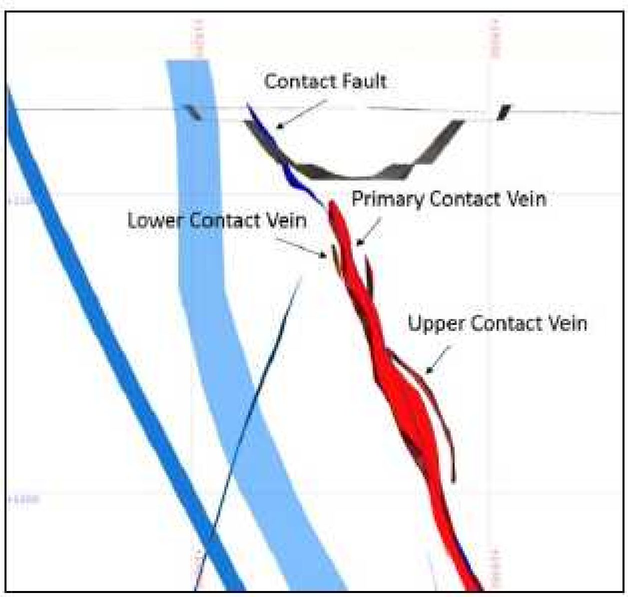


Figure 14-4: Cross section looking north of the modelled veins (primary, upper and lower)

Note: Contact fault (dark blue) and two of the eight faults modelled (light blue)

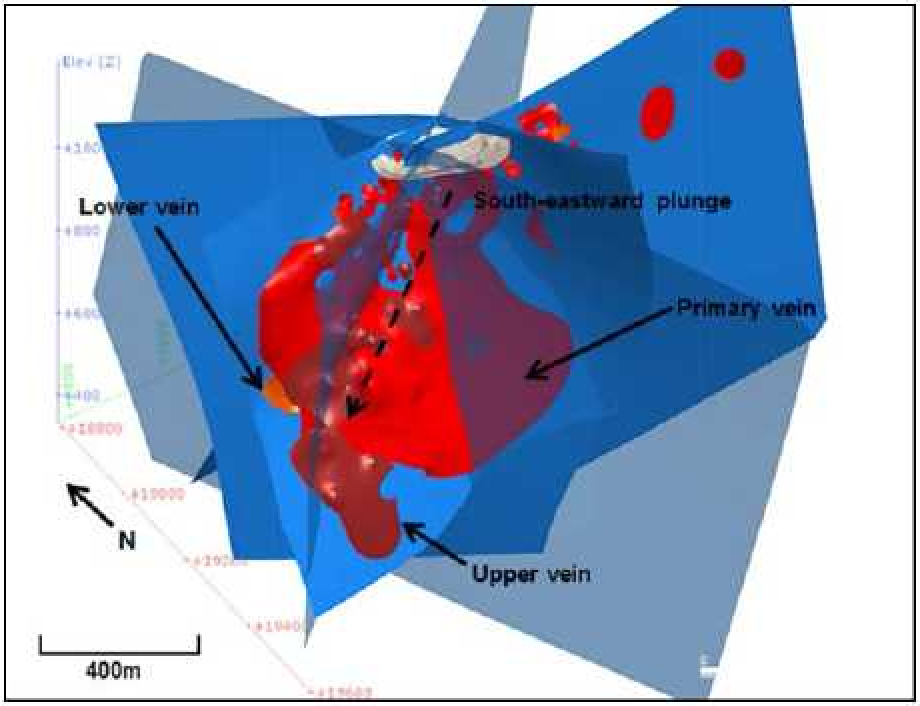


Figure 14-5: Vein architecture of the Maud Creek Deposit illustrating three primary veins and indicating a south-eastward plunge

The primary vein is strongly associated with the Sediment/Tuff contact but does not strictly follow the contact. Therefore, it has been modelled as an 'overprinting' volume on the sediment, tuff, dolerite and andesite lithology wireframes. Lithological codes BX, VEBX, VEIN or SHLE are present in most holes that intersect the contact. Where these codes are absent the vein has, in most cases, been modelled as pinched out. Where SHLE is present at, or proximal to, the contact, the intercept will carry grades similar to that of the BX, VEBX and VEIN intercepts (approximately 4 g/t Au). There are also intervals of SHLE located distal to the contact, and these do not carry grade.

For estimation purposes the upper and lower veins were combined, resulting in two domains; main vein (primary) and minor vein (upper and lower).

14.4 Grade Halo Models

Outside of and adjacent to these lithologically defined veins are many intercepts which carry similar grades to those within the vein itself. These extend up to 25 m into the hanging wall and to a lesser extent into the footwall. In addition, a greater than 0.1 g/t Au halo can be observed up to 50 m into the hanging wall and occasionally in the footwall. Excluding the veins mentioned above, the interval statistics did not indicate any grade distinction with lithology type.

Mineralization consists of two distinct zones (east and west) controlled by two north-south striking structures (Maud Creek Contact Fault and North-South Fault 1) (Figure 14-6). The western zone of the mineralization is primarily controlled by the structural contact between the footwall Sandstone and hanging wall Tuff. This zone illustrates the strongest concentration and highest grade of mineralization within the Maud Creek deposit. The fault contact strikes approximately north-south and is interpreted to have undergone reverse movement. The fault structure is filled with quartz stockwork veins; the primary host of high gold mineralization, with additional gold hosted within the surrounding wall rock. The eastern zone of mineralization is controlled by a north-south striking structure that has been inferred based on aeromagnetic interpretations (SRK, 1998) (Figure 14-6). This structure lies proximal to the contact between the Maud Creek Dolerite and Tuff units. Within this zone gold generally forms steeply dipping discrete lenses with limited continuity noted along strike.

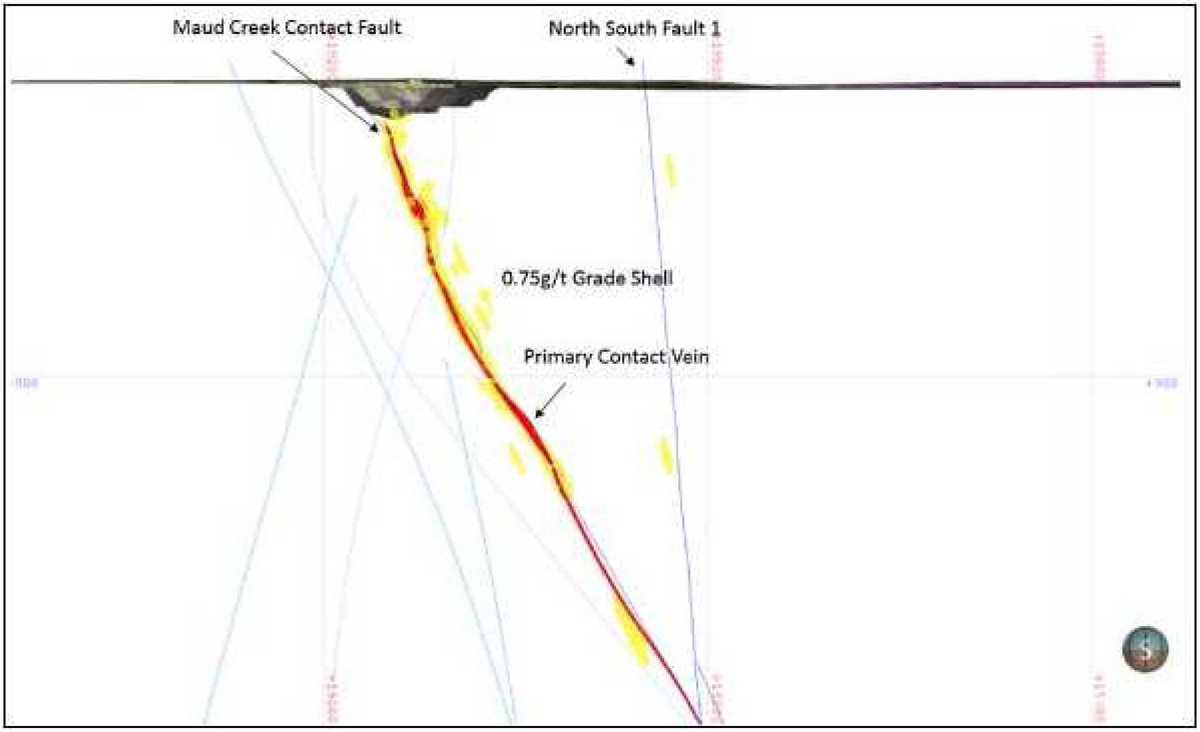


Figure 14-6: Cross section illustrating 0.75 g/t grade shell (yellow), primary contact vein (red) and fault architecture (blue)

To capture the complexity of the interactions and multiple orientations of the many faults within the deposit, grade shells generated in Leapfrog were used to model the wall rock mineralization in two stages. Although not obvious in the statistics, observation of the grade downhole suggested a sharp break at around 0.75 g/t Au. Therefore, two nested grade shells were modelled at 0.75 g/t Au and 0.1 g/t Au to be used as estimation domains. These were generated using grade from all intercepts, including those within the vein model.

In some areas of widely spaced drilling (50 m - 100m down dip) the grade shell models could not be made continuous, even though the vein had been interpreted as continuous. This is a limitation of the Leapfrog software and chosen methodology. The alternative was to manually wireframe this domain but this was considered more time consuming and less likely to capture the multiple orientations observed. There are also locations where the grade observed at the contact was too low or thin to sustain a grade shell (Figure 14-7).

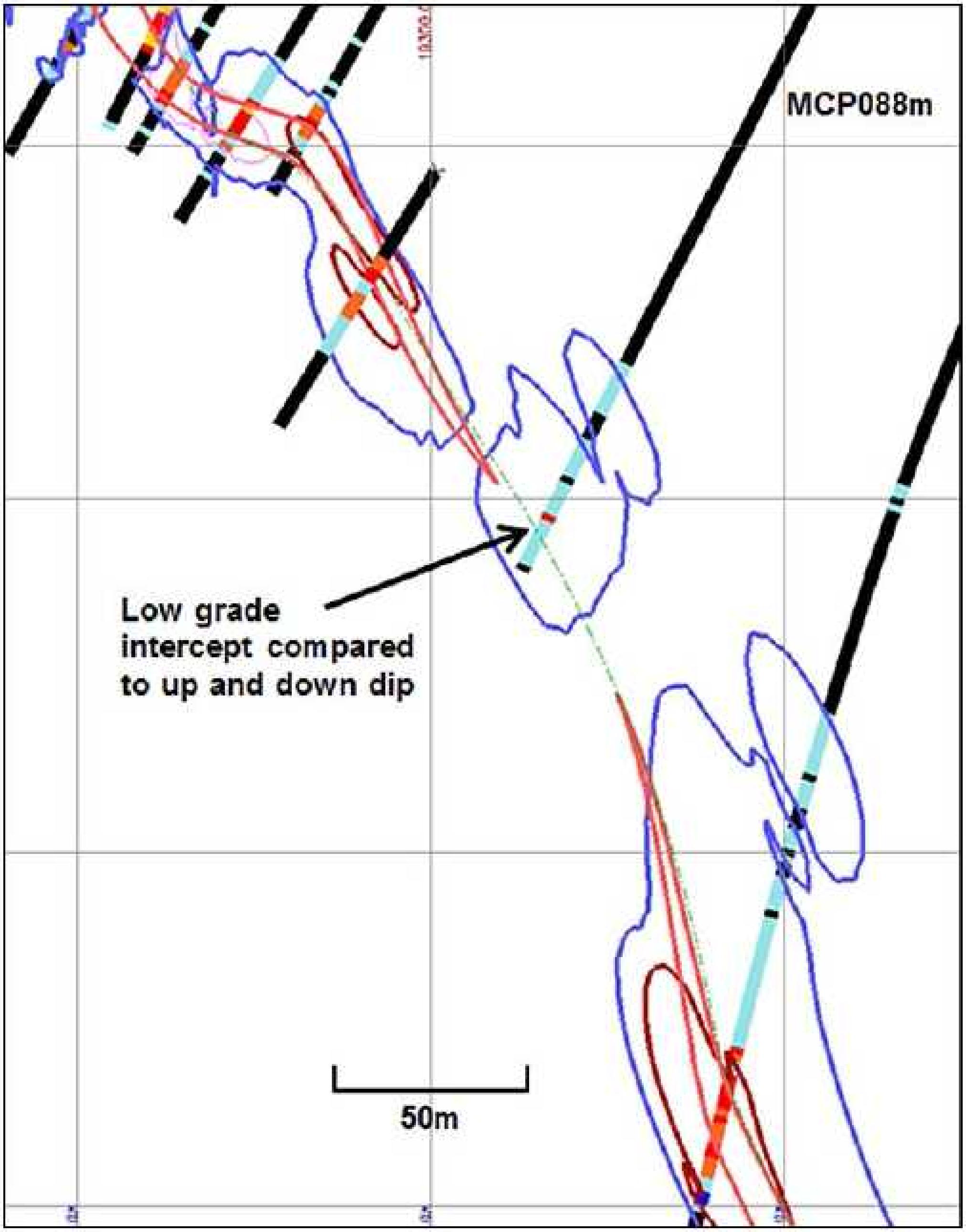


Figure 14-7: Cross section looking north (9250mN) exhibiting low grade hole MCP088 through the fault contact

14.4.1 Assumptions on non-continuity of grade adjacent to vein

The vein material (VEIN, VBX, BX or SHLE) is observed in the majority of holes that go through the contact between the Tuff and the Sediments and the model assumes Vein continuity between holes where the vein lithologies are recorded. The mineralization above 0.75 g/t Au in the footwall (Sediments) and hanging wall (Tuff) directly adjacent to the vein does not show similar continuity. Mineralization > 0.75 g/t Au may be present or absent in either the footwall or hanging wall from one hole to the next. This is observed throughout the deposit. An example is shown in Figure 14-8. MCP125 contains almost no grade in the footwall but 5 m of moderate grades in the hanging wall. The next hole down dip, MD024, contains 7 m of moderate grades in the footwall and 4 m of moderate grade in the hanging wall. The next hole down dip, MCP469 contains 2 m of grade in the footwall and 4 m of grade in the hanging wall. An assumption of continuity from hole to hole for footwall and hanging wall material, particularly in the footwall, cannot be made and this is reflected in the limited connectivity of the 0.75 g/t and 0.1 g/t Leapfrog grade halo domains.

It is also worth noting that the highest grades do not always occur in the vein and that some vein material is very low grade (Figure 14-9).

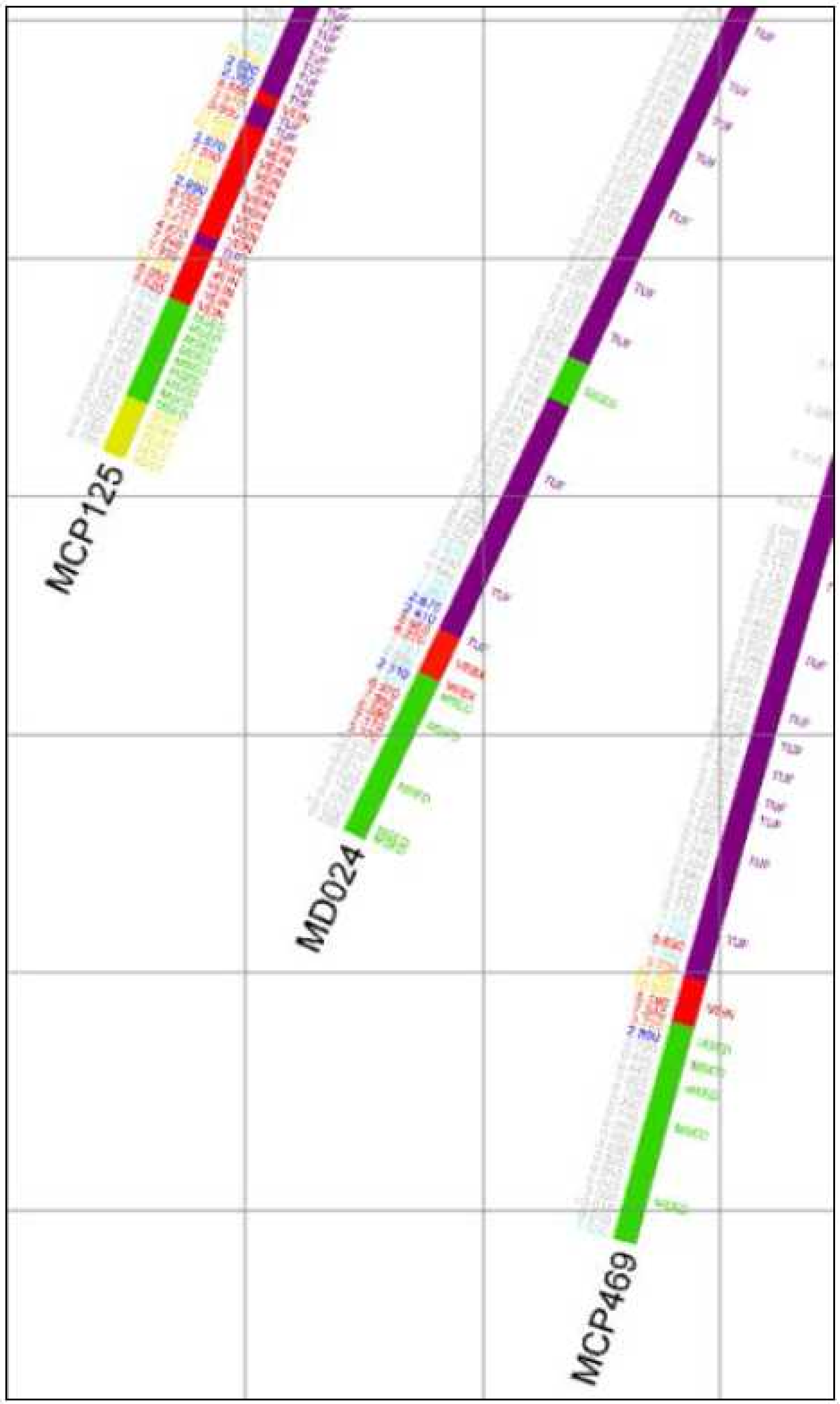


Figure 14-8: Section 9150N (20 m grid)

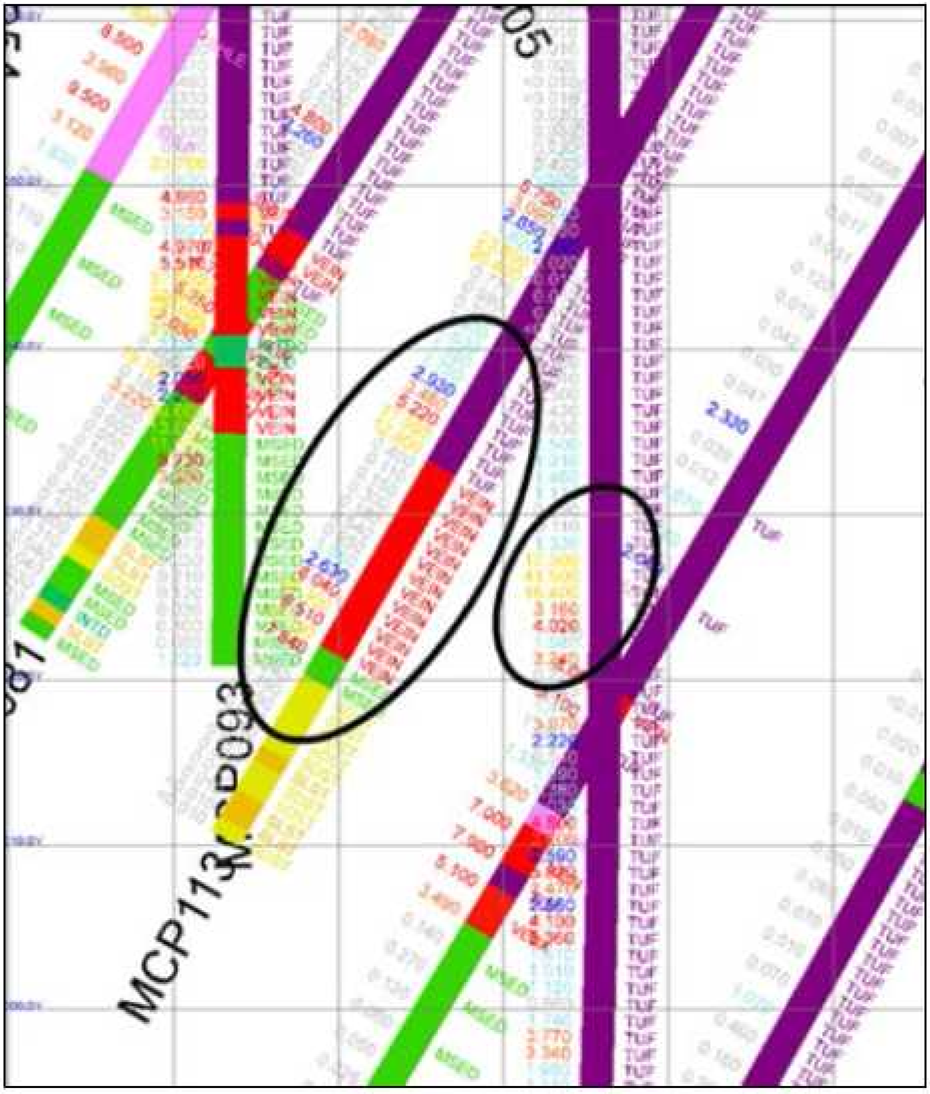


Figure 14-9: Section 9175 (20 m grid)

14.5 Domaining

The Mineral Resource was estimated into six domains as described in the previous sections:

- 0P1WEST;
- 0P75WEST;
- 0P1EAST;
- 0P75EAST;
- MAIN (major vein); and
- MINOR (minor veins).

14.6 Compositing

GEOVIA GEMS software was used to desurvey and composite the drilling. Drillholes were flagged by the different domains and then composited to 1 m intervals within the units with a minimum length of 0.01 m (Figure 14-10). This resulted in 16.5% of the composites with a length less than 1 m, however the influence of small intervals on the gold grade is limited (Table 14-2). A total of 22,449 composite samples were created.

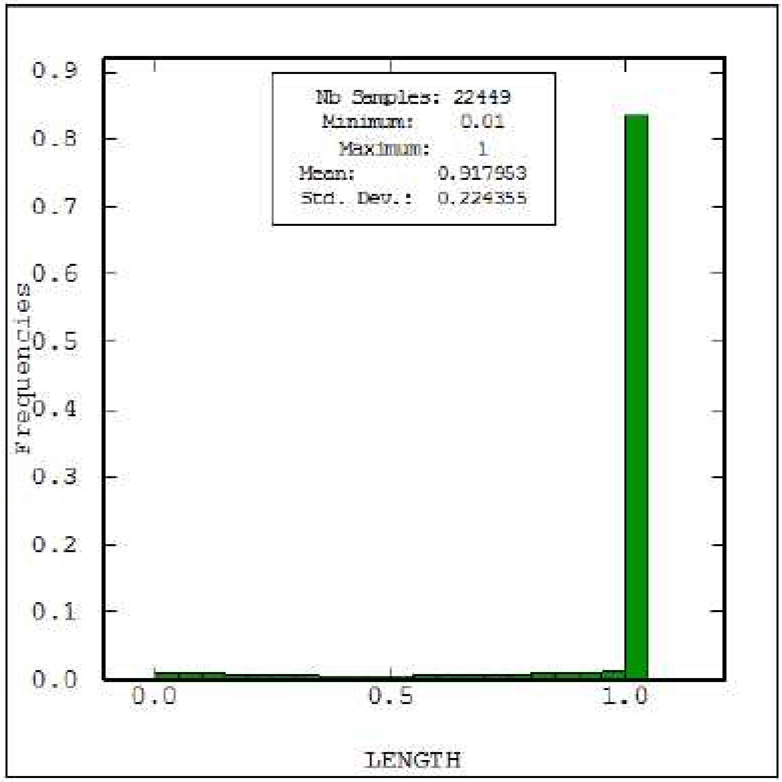


Figure 14-10: Composite length histogram

Table 14-2: Influence of the composite length on gold mean grade

Domain	Au mean grade (g/t)	Length weighted Au mean grade (g/t)	Differences
0P1WEST	0.54	0.53	1.7%
0P75WEST	4.61	4.75	-3.0%
0P1EAST	0.64	0.64	-0.6%
0P75EAST	3.71	3.86	-4.0%
MAIN	6.50	6.54	-0.5%
MINOR	5.72	6.01	-5.0%

14.7 Metallurgical Samples

The secondary elements arsenic, carbon and sulphur in the database included combined assays gathered from metallurgical studies and drill assays. The numbers of assays from metallurgical studies were minimal. Due to the compositing length the values tend to be repeated several times along a same hole creating a bias in domains with limited samples. Arsenic is an important element and the influence of the arsenic assays from metallurgical studies was analyzed. Table 14-3 indicates a fairly strong similarity between the Arsenic grades from drilling assays and metallurgical samples. Only domains with lower sample numbers indicate strong dissimilarity. Quantile-Quantile plot were also used to assess any arsenic distribution differences. The distributions were negligibly influenced by the additional metallurgical samples as shown in the MAIN domain in Figure 14-11.

Table 14-3: Comparison of arsenic assay from drilling only and including metallurgy samples

Domain	Samples Number			AS mean grade (g/t)		
	Drilling	Metallurgical	Difference	Drilling	Metallurgical	Difference
0P1WEST	5271	5365	1.8%	571	600	4.9%
0P75WEST	2149	2440	11.9%	2493	2630	5.2%
0P1EAST	1182	1188	0.5%	791	820	3.5%
0P75EAST	42	102	58.8%	3524	8311	57.6%
MAIN	641	719	10.8%	3352	3442	2.6%
MINOR	98	147	33.3%	2867	3526	18.7%

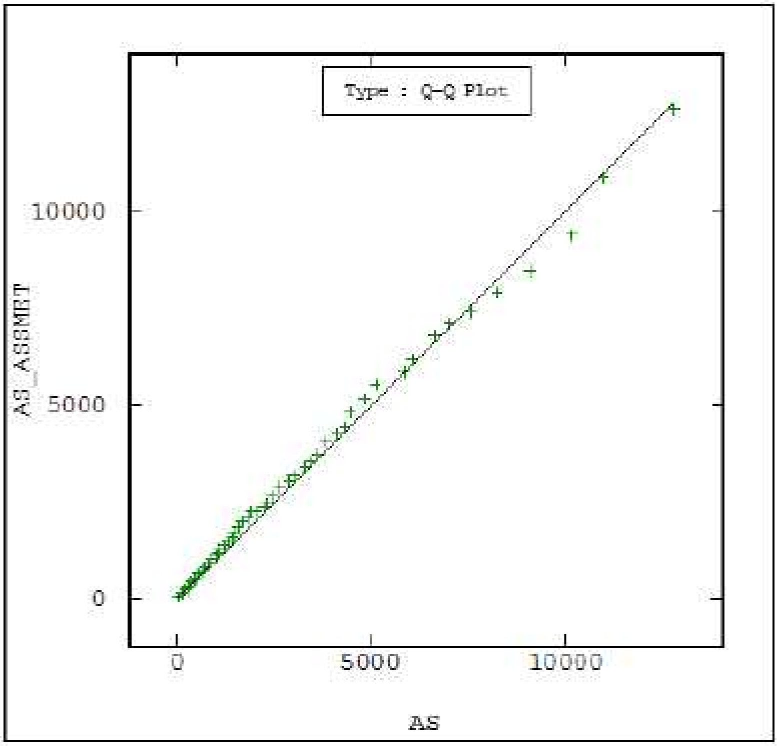


Figure 14-11: MAIN – Quantile-Quantile plot comparing arsenic from drilling only and including metallurgy samples (ASSMET)

In MINOR and 0P75EAST, the assays from the metallurgy study represent 33% and 58.8%, respectively, of the total samples. Most of these samples are located on the same hole and have been duplicated due to the compositing, as shown for the MINOR domain in Figure 14-12. Despite the differences observed in Arsenic grades the 0p75East domain and the MINOR domain all metallurgical samples were included in the drilling dataset and used for the estimation as the spatial location locations of the met samples tend to be different to the regular sample grades.

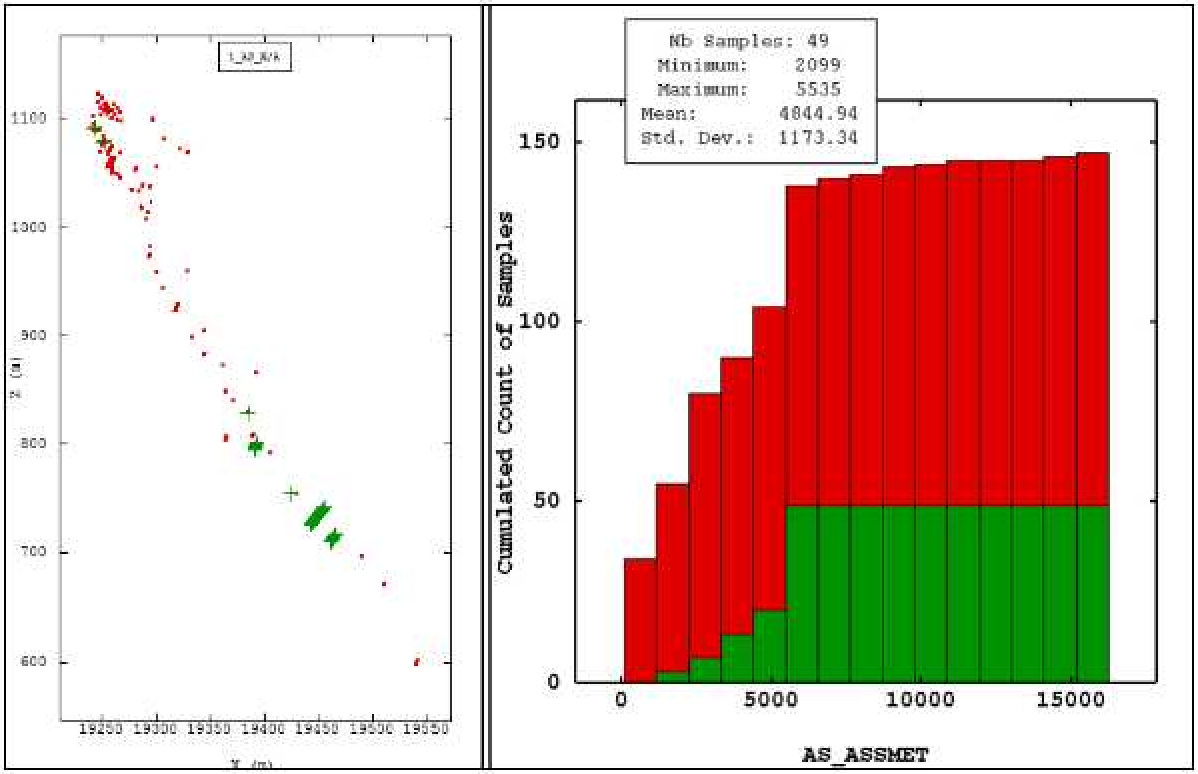


Figure 14-12: MINOR - Arsenic - Left: Plan view of data, Right: Cumulative histogram, Red: Arsenic from drilling only, Green Arsenic from the metallurgical samples

14.8 Block Model Definition

The block model size for estimation was set to 5 m x 10 m x 10 m (X, Y, Z) as this was a reasonable selective mining unit size for both open pit and underground studies, was proportional with the steep dipping north striking orientation and was a reasonable compromise to fit both the close and wiser spaced drilling.

The block model for the estimation is a proportional block model based on the domain boundaries. A sub-block model of 0.625 m x 1.25 m x 1.25 m (X, Y, Z) created in Geovia Surpac was used to visualise the blocks report the resources. The vein models are quite narrow in places and this level of sub blocking was the largest that reproduced the actual wireframe volumes within acceptable limits.

Details of the model are shown in Table 14-4.

Table 14-4: Block model properties

	X	Y	Z
Origin (m) (Lower SW corner)	19,000	8,645	400
Cell size (m)	5	10	10
Number of cells	140	140	80
Sub cell size	0.625	1.25	1.25

14.9 Grade Interpolations

The estimation strategy differed depending on the elements and domains. The elements that were estimated are Au, As, Ag, C, Cu, Sb and S. No strong correlations between the elements have been noted.

The Ordinary Kriging (OK) method of interpolation was selected to estimate the gold and arsenic grade within all six domains as these had the most complete coverage. The other elements were separately estimated using an Inverse Distance of order 2 (ID2) within the western domains and MAIN domain and a Nearest Neighbourhood (NN) method of interpolation for the eastern domains and MINOR domain. Ag, C, Cu, Sb, and S do not have complete sampling coverage and as a consequence the block estimates also do not have complete coverage of the Au estimated blocks.

The methodology used for the estimation consisted of:

- Performing cell declustering tests;
- Studying the influence of outliers;
- Validating the choice of using samples from all drillhole type;
- Studying the spatial variability for gold and arsenic (variography);
- Defining an estimation strategy; and
- Validating the estimation.

14.10 Declustering

Drilling includes various types of holes with unequal spacing, the highest drilling density corresponding to the open pit area and near surface. A cell declustering test was performed using a grid specific range specific to each domain and element. The declustering results are shown in Table 14-5, Table 14-6 and Table 14-7. A positive difference means that the declustered grade is lower than the undeclustered grade.

Table 14-5: Declustering results in the vein domains

Elements	MAIN			MINOR		
	Cell Size (m)	Declustered Mean (g/t)	Differences	Cell Size (m)	Declustered Mean (g/t)	Differences
AU	40x40x40	5.67	13%	90x90x90	3.26	43%
AS	90x90x90	2,993.1	13%	90x90x90	2,707.9	23%
AG	60x60x60	3.42	-0.8%	50x50x50	2.84	12.5%
CU	60x60x60	176.3	2.9%	50x50x50	88.9	29.9%
C	60x60x60	1.62	1.4%	50x50x50	1.04	-2.6%
SB	60x60x60	19.4	14.4%	50x50x50	29.95	18.1%
S	60x60x60	1.43	-3.1%	50x50x50	1.8	-5.1%

Table 14-6: Declustering results in the western domains

Elements	0P1WEST			0P75WEST		
	Cell Size (m)	Declustered Mean (g/t)	Differences	Cell Size (m)	Declustered Mean (g/t)	Differences
AU	40x40x40	0.47	13%	40x40x40	3.11	32%
AS	50x50x50	817.8	-36%	30x30x30	2,564.2	3%
AG	60x60x60	1.79	13.5%	60x60x60	2.99	28.2%
CU	60x60x60	91.2	4.2%	60x60x60	112.4	5.5%
C	60x60x60	1.75	13.3%	60x60x60	1.59	0.4%
SB	60x60x60	20.0	14.8%	60x60x60	26.2	12.6%
S	60x60x60	1.07	-21.1%	60x60x60	1.32	1.5%

Table 14-7: Declustering results in the eastern domains

Elements	0P1EAST			0P75EAST		
	Cell Size (m)	Declustered Mean (g/t)	Differences	Cell Size (m)	Declustered Mean (g/t)	Differences
AU	40x40x40	0.55	14%	40x40x40	3.77	-2%
AS	40x40x40	991.2	-21%	50x50x50	1,0024.8	-21%
AG	30x30x30	0.91	-4.5%	30x30x30	8.23	0.0%
CU	30x30x30	60.1	2.8%	30x30x30	91.6	0.0%
C	30x30x30	1.89	-0.2%	30x30x30	-	-
SB	30x30x30	33.3	3.1%	30x30x30	62.5	0.0%
S	30x30x30	1.24	-69.6%	30x30x30	1.80	-6.4%

14.11 Outliers

The element grades have a skewed distribution within each domain as shown by the coefficient of variation shown in, Table 14-8 and Table 14-9. The histograms show a long tail for high grades. In the eastern domains and Minor domain, the grade distributions of the elements other than gold and arsenic are not well represented due to a limited number of samples.

MAIN domain, Gold grade distribution is strongly skewed with a coefficient of variation of 5.2. Significant high grade samples are skewing the Gold grade distribution. In particular, two samples have been identified above 200g/t which are relatively isolated from the main distribution and have a strong impact on the statistics and variography (Figure 14-13). They are located approximately 425 m below surface at the edge of the domain (Figure 14-14).

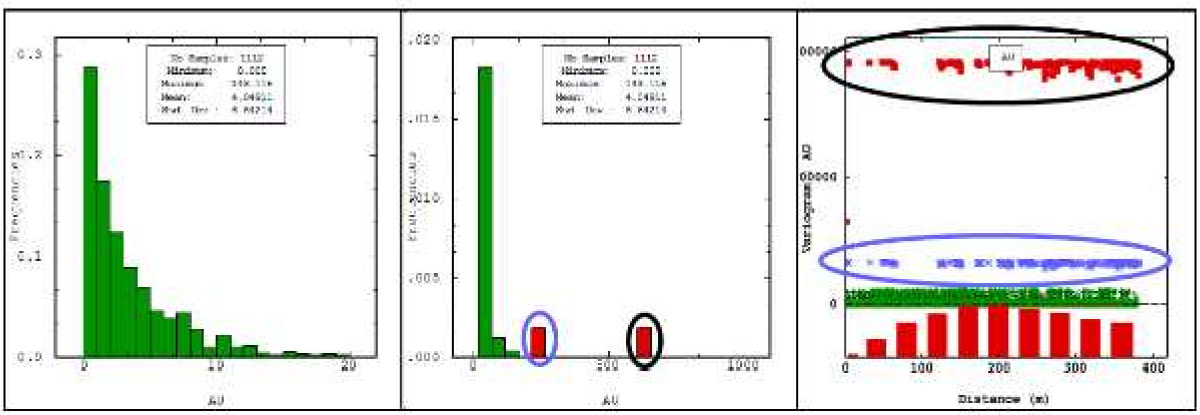


Figure 14-13: MAIN - Left and Middle Au histogram: from 0 - 20g/t and from 20 - 1000 g/t and Right: Variogram cloud, Red and blue: outliers

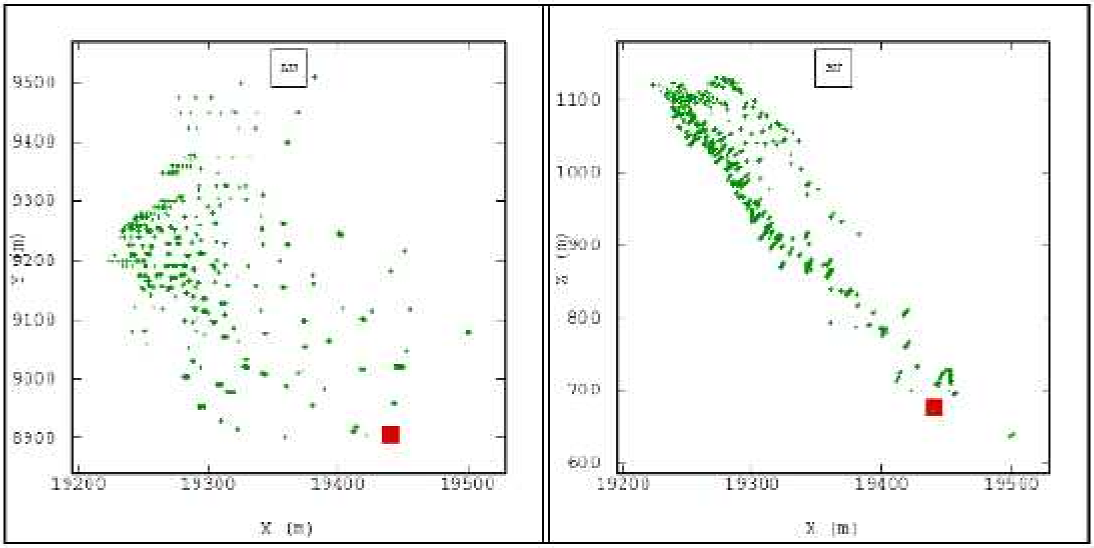


Figure 14-14: MAIN – Au samples location – Left: XoY view, Right: XoZ view
Red: outliers

Further analysis was undertaken for each element within each domain. According to the influence of the outliers on the variography and their locations, a top cut was applied or not. When the outliers have a strong impact, a top cut was applied. Values above the top cut were replaced by the top cut threshold. Table 14-8 and Table 14-9 summarise the treatment of the outliers per domain.

Table 14-8: Outlier treatment in veins domains

Elements	MAIN			MINOR		
	# Samples	Declustered Coeff. Variation	Top cut threshold (g/t)	# Samples	Declustered Coeff. Variation	Top cut threshold (g/t)s
AU	1,114	5.2	50	4,615	1.7	50
AS	719	0.9	-	147	0.8	10,000
AG	147	1.0	-	28	0.8	6.5
CU	148	2.5	-	28	1.0	300
C	43	0.7	2	31	0.2	2
SB	57	1.0	-	9	0.8	50
S	121	0.7	-	47	0.4	2.1

Table 14-9: Outlier treatment in western domains

Elements	0P1WEST			0P75WEST		
	# Samples	Declustered Coeff. Variation	Top cut threshold (g/t)	# Samples	Declustered Coeff. Variation	Top cut threshold (g/t)s
AU	11,091	2.5	50	4,615	1.7	-
AS	5,365	1.7	-	2,440	1.1	-
AG	1,369	2.2	-	499	2.0	-
CU	1,443	2.0	-	505	1.5	-
C	96	0.7	-	229	0.8	-
SB	298	0.8	-	233	0.9	-
S	263	0.9	-	388	0.7	-

Table 14-10: Outlier treatments in eastern domains

Elements	0P1EAST			0P75EAST		
	# Samples	Declustered Coeff. Variation	Top cut threshold (g/t)	# Samples	Declustered Coeff. Variation	Top cut threshold (
AU	3,408	3.3	40	386	1.1	-
AS	1,188	2.5	-	102	0.9	12,000
AG	192	1.2	5.5	11	0.7	-
CU	190	1.1	-	11	0.4	-
C	31	0.2	2	0	-	-
SB	73	0.4	-	11	0.4	-
S	46	1.0	1.9	71	0.5	1.9

14.12 Drillhole Types

The drilling dataset used for the estimation consists of 677 drillholes including 1 RAB hole for 1.91 m composite length, 93 DD holes totalling 6107.8 m composite length and 583 RC holes totalling 14,497 m composite length. 256 RC holes are from grade control RC drillholes and are called RCGC. Figure 14-15 shows in red the location of the RCGC on the projected northing section.

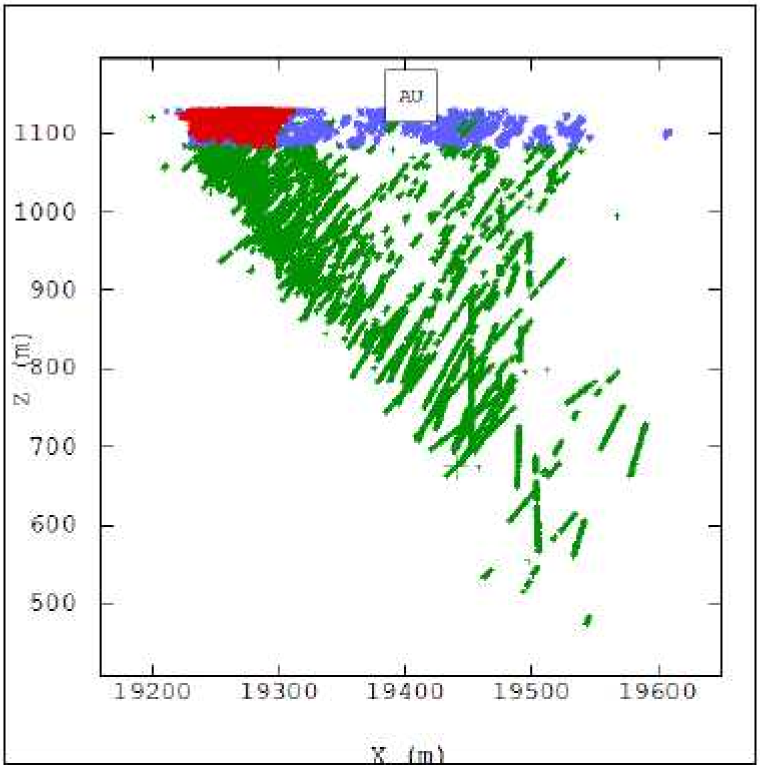


Figure 14-15: AU samples collected in the RC drillholes, Red: RCGC, Blue: RC samples above 1,084 mRL and Green: remaining RC

The statistical comparison of drillhole types was undertaken using weighted composite data and outlier treatment.

Figure 14-16 compares the differences Au mean grade between RC above 1,084 mRL and RCGC. RCGC were not found in the eastern domains. The Au mean grade compares relatively well between both drilling types at the exception of MINOR domain. In MINOR domain, Figure 14-17 indicates that the Au distribution from RC samples above 1,084 mRL is not well defined after 24 g/t compares to the distribution within the RCGC samples. SRK considers that there is no bias between the RC and RCGC samples.

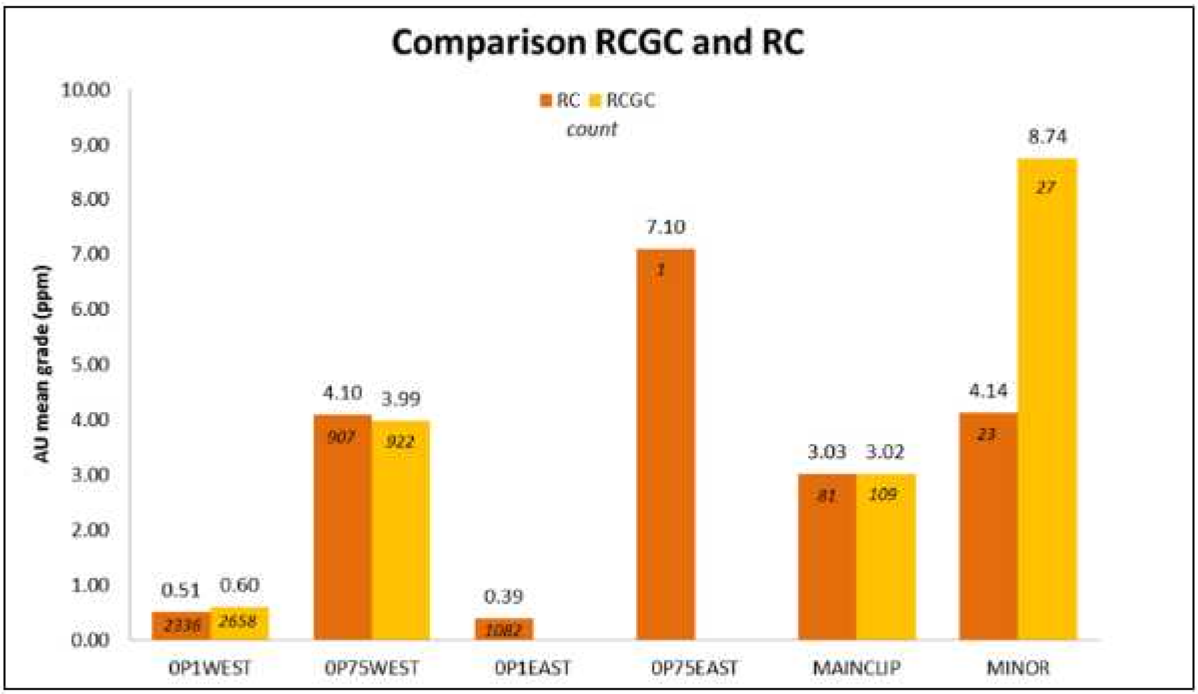


Figure 14-16: Comparison of weighted and top-cut Au mean grade between RC and RCGC above 1084 mRL

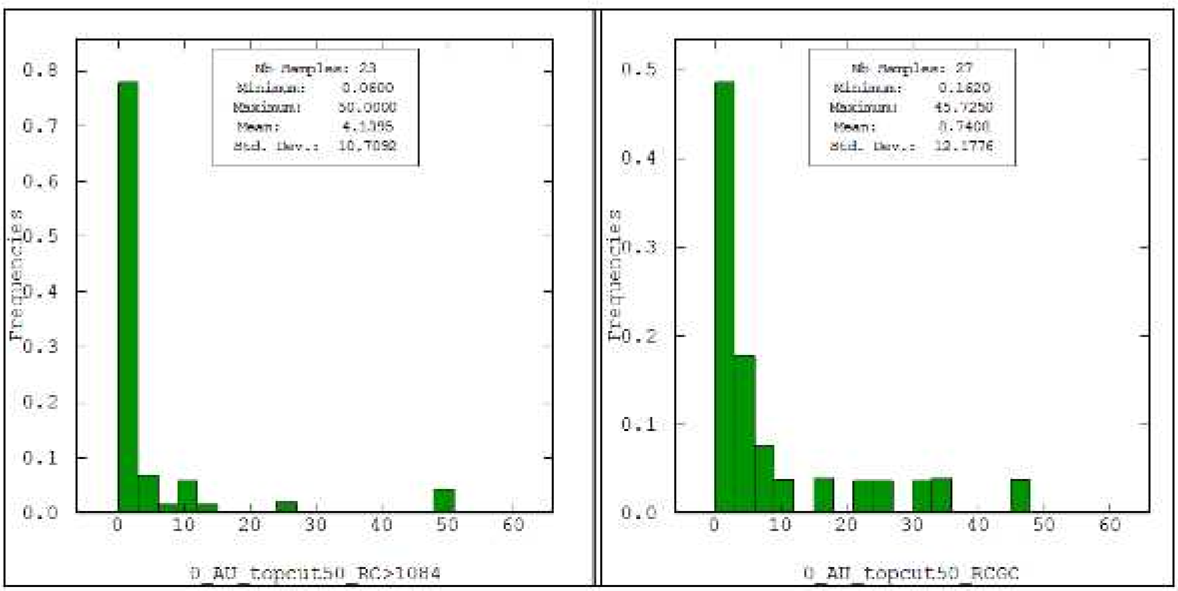


Figure 14-17: MINOR - Au distribution - Left: RC above 1,084 m, Right: RCGC

Samples from RCGC drillhole type were combined with all the samples from RC drillhole type. From now on any reference to RC drillhole type includes RCGC drillhole type.

Figure 14-18 compares the Au mean grade between the DD and RC within each domain. However the differences are less than 10% in most domains, the statistics fluctuate with the number of samples and are influenced by high grade values. Two third of the composite samples are from RC samples. Small domains with low number of samples tend to have more discrepancies between data types. SRK considers that there is no bias between the RC and DD samples.

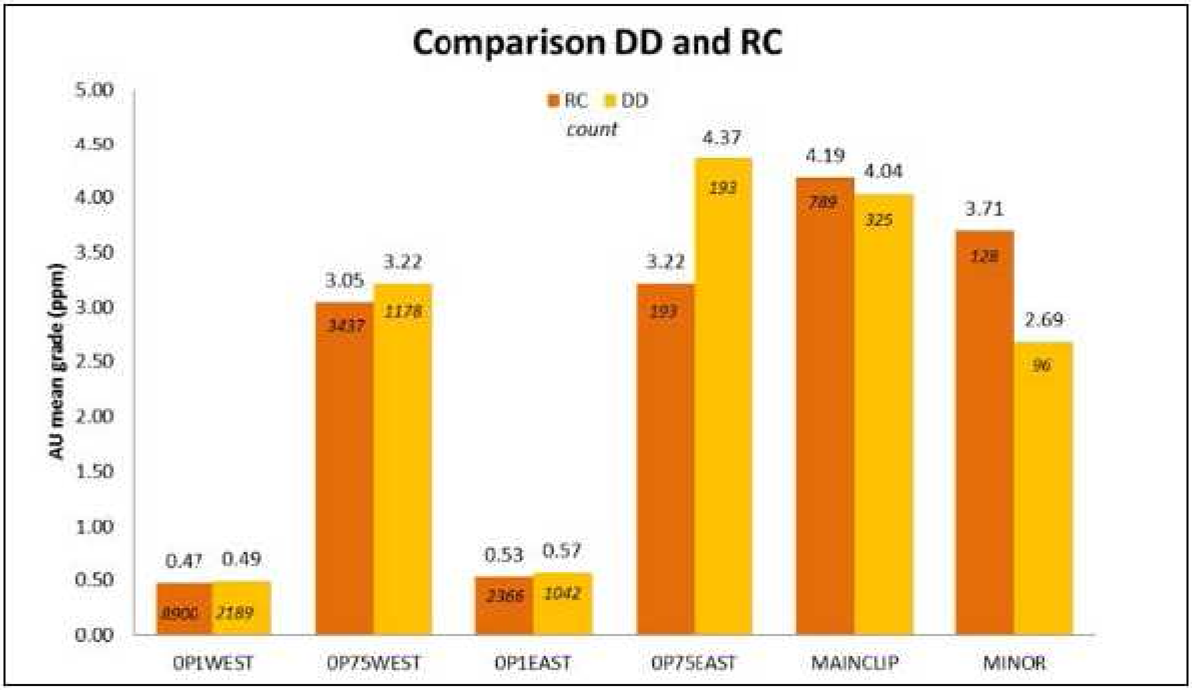


Figure 14-18: Comparison of unweighted Au mean grade per drilling type

The samples from DD, RC and RCGC were used for the estimation.

14.13 Summary Statistics

This section compares the mean gold grade, declustered grade and top-cut grade within each domain (Table 14-11).

Table 14-11: Comparison of mean gold grade (g/t) within each domain

Domains	AU mean grade (g/t)		
	Unweighted	Weighted	Top cut applied
0P1WEST	0.54	0.47	0.47
0P75WEST	4.61	3.11	3.11
0P1EAST	0.64	0.55	0.54
0P75EAST	3.71	3.77	3.77
MAIN	6.50	5.67	4.13
MINOR	5.72	3.26	3.20

14.14 Variography

Spatial variability was studied for the two main elements gold and arsenic within each domain. The variography of gold and arsenic at Maud Creek was completed using the Isatis software, produced by Geovariance.

The choice of variogram directions were constrained by a combination of drilling data, geological continuity and domains. The structures observed were poor in most domains. To improve the variography study, the Gold grade was transformed into a Gaussian value within each domain while the arsenic was only transformed into a Gaussian value in domain 0P1EAST and 0P1WEST. The transformation was completed using a punctual Gaussian anamorphosis. The number of Hermite polynomial was adjusted according to the domain and element, varying between 30 and 70.

The variogram was computed on the 1 m Gaussian composites within each domain separately.

A model was fitted to the Gaussian variogram and back transformed to a model representing the composite data.

The parameters for the variogram model of the gold and arsenic for all the domains are given in Table 14-12 and Table 14-13. As expected, the Au element is fairly variable with a nugget that accounts for at least 44% of the total sill.

In MAIN domain, the best model is reasonably well described by two spherical structures and a nugget for both elements. The nugget for the Au variogram account for 74% of the total sill while the nugget for the arsenic variogram account for only 4%. The first structure is relatively short. Figure 14-19 shows the variogram fitting within the MAIN domain for gold and arsenic.

Table 14-12: Variogram model characteristics for Au

Domains	Rotation Geologist Plane			Structures	Sills	Range		
	AZI	DIP	PITCH			U	V	W
0P1WEST	0	67	161	Nugget	0.97			
				Spherical 1	0.44	30	30	5.6
0P75WEST	0	67	90	Nugget	16.42			
				Spherical 1	4.97	35	35	2.8
				Spherical 2	3.92	35	35	12
				Spherical 3	2.31	90	35	12
0P1EAST	0	65	95	Nugget	2.7			
				Spherical 1	0.62	20	20	7
0P75EAST	Omni-directional			Nugget	14.23			
				Spherical 1	3.48	110	110	-
MAIN	0	65	116	Nugget	625.12			
				Spherical 1	162.15	40	40	20
				Spherical 2	60.73	160	100	20
MINOR	0	65	116	Nugget	13.35			
				Spherical 1	9.46	40	40	20
				Spherical 2	7.79	160	100	20

Table 14-13: Variogram model characteristics for Arsenic

Domains	Rotation (degree) Geologist Plane			Structures	Sills (g/t)	Range (m)		
	AZI	DIP	PITCH			U	V	W
0P1WEST	0	100	-180	Nugget	266,079			
				Spherical 1	456,877	40	20	10
				Spherical 2	402,390	200	80	50
0P75WEST	0	67	180	Nugget	1,000,000			
				Spherical 1	5,200,000	25	40	9
				Spherical 2	2,000,000	140	40	40
0P1EAST	0	70	-180	Nugget	1,840,481			
				Spherical 1	2,526,700	50	40	35
				Spherical 2	2,001,163	100	40	35
0P75EAST	Omni-directional			Spherical 1	11,050,000	19	19	-
MAIN	0	85	170	Nugget	300,000			
				Spherical 1	5,500,000	35	35	15
				Spherical 2	2,000,000	100	80	15
MINOR	Omni-directional			Nugget	800,000			
				Spherical 1	4,000,000	40	40	40

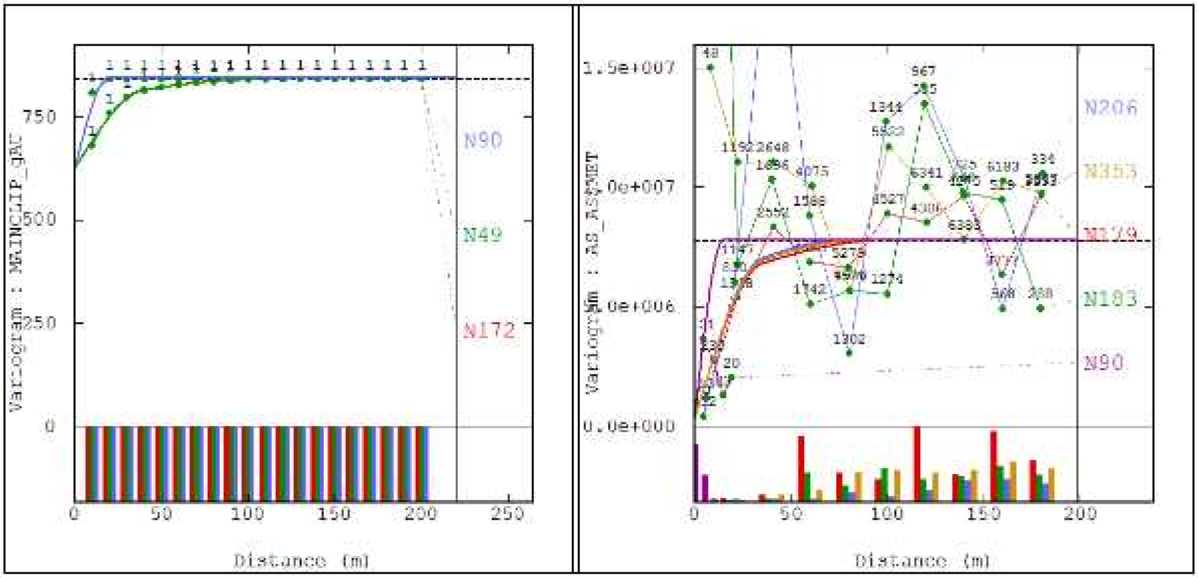


Figure 14-19: MAIN variogram fitting – left: Au, Right: As

14.15 Estimation

14.15.1 Gold and Arsenic

The Ordinary Kriging (OK) method was selected to estimate gold and arsenic within each domain. Domain boundaries are considered hard, with two exceptions.

- Composites from the MAIN were allowed to be used in the estimation of the MINOR domain (soft boundary, although the domains are not in direct contact).
- To address the potential under call in ounces where the 0.75 g/t Leapfrog grade shells did not connect, a two stage process was used to estimate the 0.1 g/t Au domain. Firstly the 0.1 g/t Au domain was estimated using a hard boundary constrained between 0.1 and 0.75 g/t Au material. Secondly, and overriding the first estimation, a soft boundary was estimated using all of the material located outside the main vein (primary). A tight search ellipse orientated perpendicular to the dip direction was employed, to ensure only those blocks within the 0.1 g/t Au domain, directly adjacent to the main vein, were informed during the second estimation. (The overall effect of this methodology was to increase the 0.1 g/t Au domain from an average grade of 0.48 g/t to 0.7 g/t Au, with some extra 140 koz at zero cut of.)

Identified outliers were replaced by the top cut when they exceeded a threshold distance of 15 m from the block centroid. Within 15m the full uncut value was used.

The estimation was done using one neighbourhood search size specific to each domain. The maximum number of samples selected for the estimation is a compromise between the best local smoothed estimates, using many samples to optimise the slope of regression minimise negative weights and the best global grade and tonnage curve, de-smoothed, estimate using fewer samples. Details of the search ellipsoids for gold are shown in Table 14-14 and Table 14-15. Details of the search ellipsoids for arsenic are shown in Table 14-16. Note that the maximum number of samples is given by the number of sectors multiplied by the optimum number of sample per sector.

Table 14-14: Neighbourhood parameters – Au estimated by OK

Domains	0P1WEST	0P75WEST	0P1EAST	0P75EAST	MAIN	MINOR
Rotation						
X Angle	0	0	0	0	0	0
Y Angle	67	67	65	65	67	67
Z Angle	161	90	95	95	116	116
Search Ellipsoid						
X max	400	400	400	400	600	600
Y max	400	400	400	400	600	600
Z max	200	200	200	200	300	300
Number of sectors	8	8	10	8	1	1
Minimum samples	8	8	8	8	8	8
Optimum samples per sector	4	4	3	4	32	32
Minimum distance between data				0.7		
Cut-off						
Threshold	50		40		200	50
Distance	15		15		15	15

Table 14-15: Neighbourhood parameters – Au estimated by OK – 0P1WEST soft boundary

Domains	0P1WEST (Soft boundary)
Rotation	
X Angle	0
Y Angle	67
Z Angle	161
Search Ellipsoid	
X max	400
Y max	400
Z max	10
Number of sectors	8
Minimum samples	4
Optimum samples	4
Minimum Number samples per line	2
Cut-off	
Threshold	50
Distance	15

Table 14-16: Neighbourhood parameters – AS estimated by OK

Domains	0P1WEST	0P75WEST	0P1EAST	0P75EAST	MAIN	MINOR
Rotation						
X Angle	0	0	0	0	0	0
Y Angle	100	67	70	65	85	67
Z Angle	-180	90	-180	95	170	116
Search Ellipsoid						
X max	400	400	400	400	600	600
Y max	400	400	400	400	600	600
Z max	200	200	200	200	300	300
Number of sectors	8	4	4	8	4	1
Minimum samples	40	1	1	40	1	40
Optimum samples per sector	10	8	20	10	10	400
Minimum distance between data				0.7		
Maximum Distance without sample	50				90	
Cut-off						
Threshold				12000		10000
Distance				15		15

14.15.2Other elements

The Inverse Distance of order 2 (ID2) method was selected to estimate the other elements in the western domains and MAIN while the Nearest Neighbourhood (NN) method was selected to estimate in the eastern domains and MINOR. ID2 and NN characteristics are as follow:

- Domain boundaries are considered hard, at the exception of the contact between MINOR and MAIN when estimating in the MINOR domain and the eastern domains.
- Identified outliers were replaced by the top cut if outside the threshold distance.
- The search orientation and size were similar for all domains. The number of data used for the estimation is limited by the neighbourhood parameters. Details of the search ellipsoids are shown in Table 14- 17.
- Due to the limited number of data, a restriction on the distance without samples was applied meaning that not all the block cells were informed during the ID2 or NN estimation.

Table 14-17: Neighbourhood parameters

Domains	0P1WEST	0P75WEST	0P1EAST	0P75EAST	MAIN	MINOR
Rotation						
X Angle	0					
Y Angle	90					
Z Angle	180					
Search Ellipsoid						
X max	400					
Y max	400					
Z max	200					
Number of sectors	4					
Minimum samples	10					
Optimum samples	10					
Maximum Distance without sample	60	60	30	30	60	50

14.16 Density

There are three periods of density sampling over the life of the project as recorded in the density database supplied by Newmarket Gold. Around 1991 Places made 43 measurements from the WD series holes from 0.1 to 0.3m lengths of core. Around 1995 Kalmet took 481 measurements from the MD series holes from 0.1m to 0.3m intervals of core. In 2011 Newmarket Gold took 2.145 measurements from 0.1m lengths of core from the MC series holes. The Newmarket Gold measurements used some half core and some full core.

There are a total of 2.669 density measurements available, 740 of which are in mineralization (as defined by the estimation domains). Oxide states show significant differences in density (Table 14-18 and Table 14-19). There is no practical difference in densities within the fresh material for lithology or estimation domain (Table 14-20, Table 14-21 and Table 14-22). The densities shown in Table 14-18 have been used to inform both the resource and waste model.

Table 14-18: Density statistics by oxide state all data

Oxide state	Count	Density
1 (completely oxidised)	3	2.11
2 (oxide / transition)	35	2.60
3 (transition)	53	2.72
4 (transition/ fresh)	39	2.77
5 (fresh)	2538	2.80

Table 14-19: Density statistics by oxide state in all estimation domains

Oxide state	Count	Density
1 (completely oxidised)	2	2.10
2 (oxide / transition)	23	2.57
3 (transition)	16	2.65
4 (transition/ fresh)	3	2.77
5 (fresh)	696	2.79

Table 14-20: Density statistics by estimation domain within all estimation domains in Fresh rock

Domain	Count	Density
0.1 halo	249	2.80
0.75 halo	327	2.80
Veins	120	2.78

Table 14-21: Density statistics by lithology in Fresh rock all data

Lithology	Count	Density
Tuff	1016	2.79
Sediments	640	2.59
Minor Vein	30	2.76
Major Vein	90	2.78
Dolerite	762	2.85

Table 14-22: Density statistics by lithology in estimation domains in in Fresh rock

Lithology	Count	Density
Tuff	387	2.81
Sediments	160	2.77
Minor Vein	31	2.76
Major Vein	90	2.78
Dolerite	29	2.82

14.17 Validation

Following the estimation, the final model was reviewed and validated. The estimation validation consists of statistical comparison, visual comparison and swath plots. The estimation validation of the OK includes also the review of the kriging output variables such as the regression slopes.

The regression slopes of the Au estimates were relatively poor with a high number of values lower than 0.5. Figure 14-20 shows the results for the gold and arsenic within the MAIN domain.

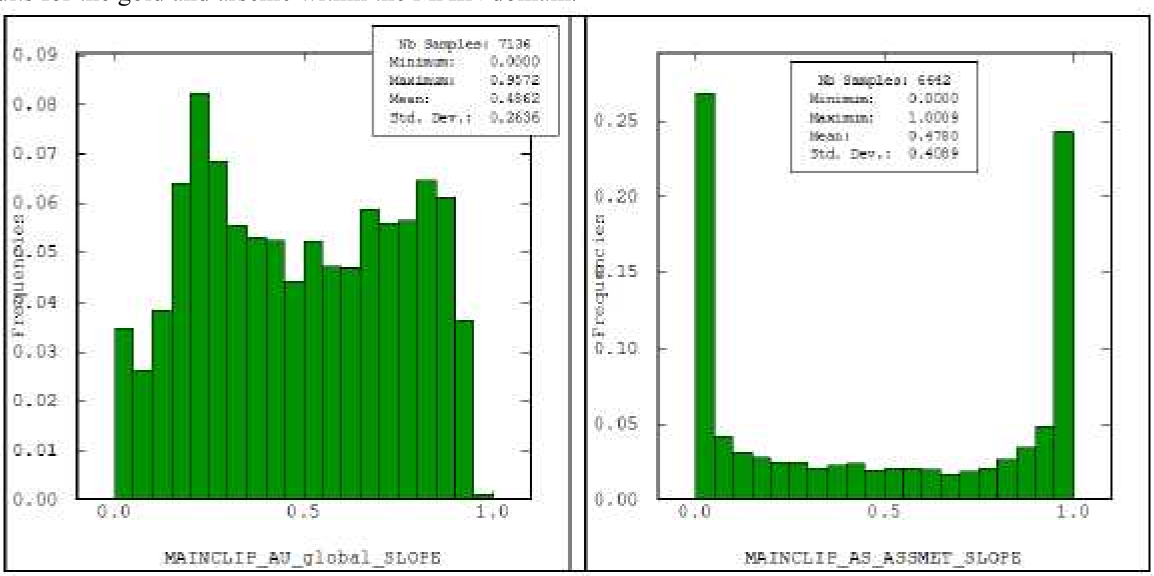


Figure 14-20: Regression slope histogram for Au estimates (left) and AS estimates (right)

The gold estimation by OK was re-run by increasing the maximum allowable number of data used in the search neighbourhood to 400. Increasing the maximum number of samples used for estimation in the neighbourhood definition improves the local block OK performance and results in lower slopes of regression overall (Figure 14-21) but over smooths the block distribution such the grades and tonnages at higher cut-offs are not realistic (Figure 14-23). For identification purposes in this section of the report, the check run OK is referred to Local OK while the gold estimation is called Global OK. Table 14-23 compares the global statistics between the composite top cut data and the two gold estimates. Global statistics showed reasonable good correlation at zero cut-off between the estimates and the data, slightly better for the Global OK estimates.

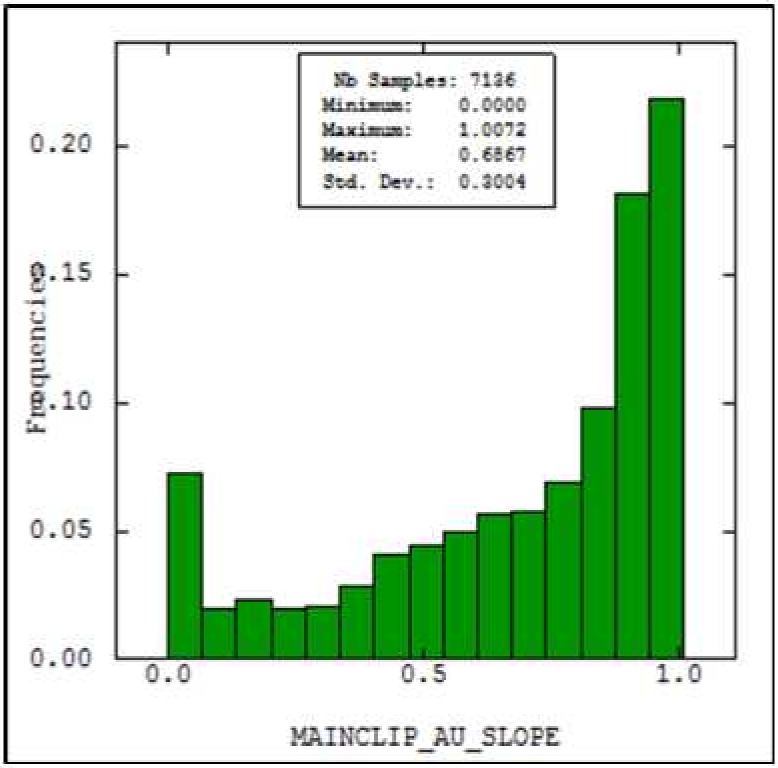


Figure 14-21: Regression slope histogram for Main Au Local check estimate

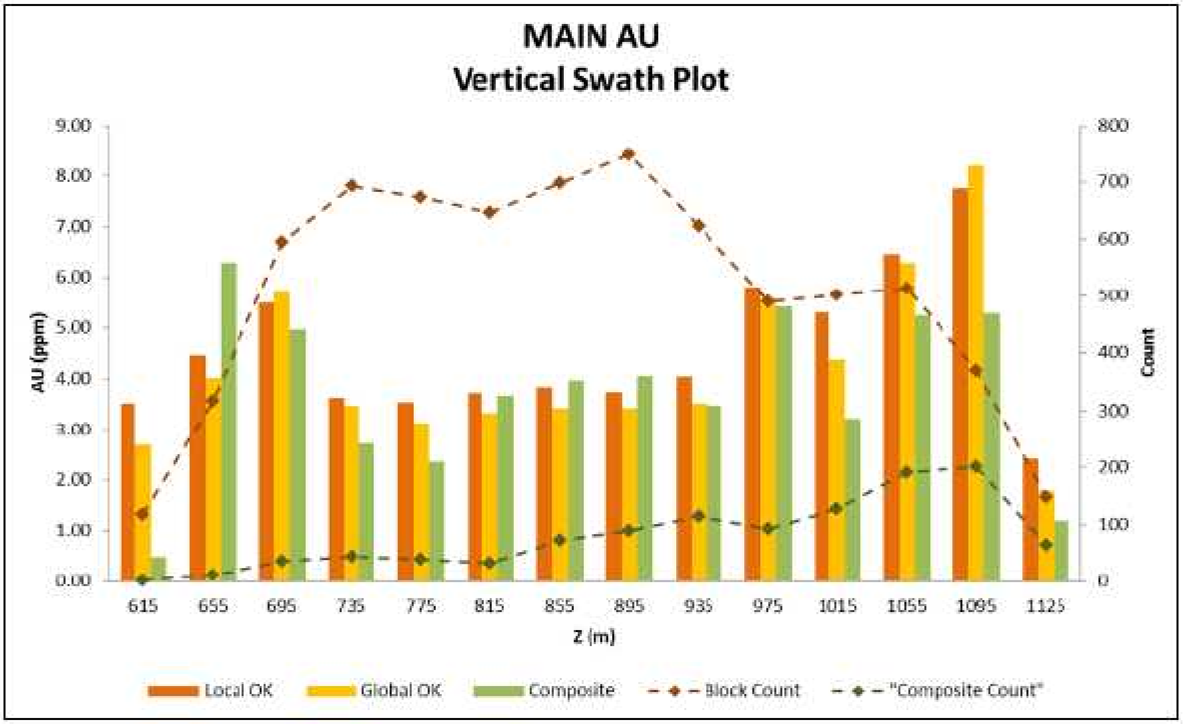
Table 14-23: Comparison of declustered top cut Au composite and estimates per domain

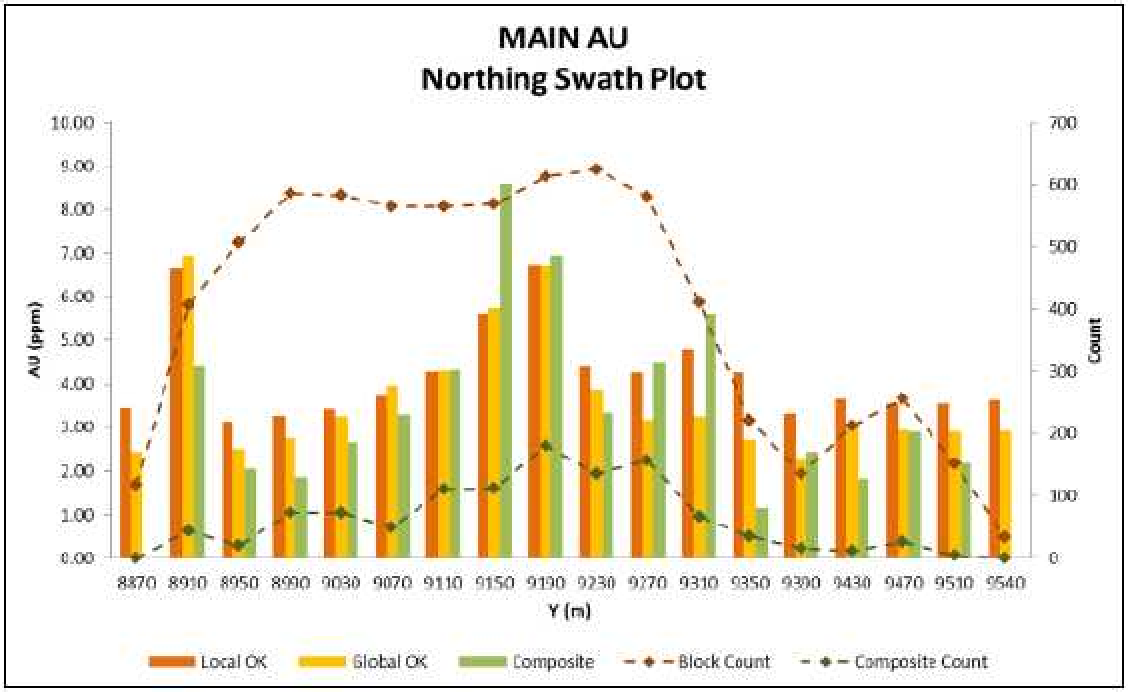
Domains	Variable	# Samples/ blocks	Minimum	Maximum (Cut max)	(Declustere d) Mean	% Difference
0P1WEST	Composite	11,091	0.001	58.18 (50)	0.47	
	Local OK	28,831	0.13	7.13	0.48	2.1
	Global OK	28,831	0.11	7.14	0.46	-2.1
0P75WEST	Composite	4,615	0.05	282.83	3.11	
	Local OK	8,650	0.99	25.70	3.33	7.1
	Global OK	8,650	0.95	28.61	3.34	7.4
0P1EAST	Composite	3,408	0.01	76.07 (40	0.54	
	Local OK	12,592	0.18	6.03	0.58	7.4
	Global OK	12,592	0.11	7.55	0.54	0.0
0P75EAST	Composite	386	0.01	25.38	3.77	
	Local OK	992	1.71	10.91	4.06	7.7
	Global OK	992	1.29	11.13	3.98	5.6

Domains	Variable	# Samples/ blocks	Minimum	Maximum (Cut max)	(Declustere d) Mean	% Difference
MAIN	Composite	1,114	0.005	619.43 (50)	4.13	
	Local OK	7136	0.63	91.87	4.32	4.6
	Global OK	7,136	0.06	103.36	4.30	4.1
MINOR	Composite	224	0.01	79.67 (50)	3.20	
	Local OK	1,880	0.228	24.874	3.65	14.1
	Global OK	1,880	1.499	22.599	3.96	23.8

At 0.0 g/t cut-off grade

Within the MAIN domain, 40 m swath plot in X, Y and Z directions showed a reasonably good correlation within area well sampled (Figure 14-22). In areas with many composite samples, the OK has smoothed the grade while in area with limited composite samples the OK tends to overestimate. The Local OK is mostly over-estimating the grade compared to the Global OK approach.





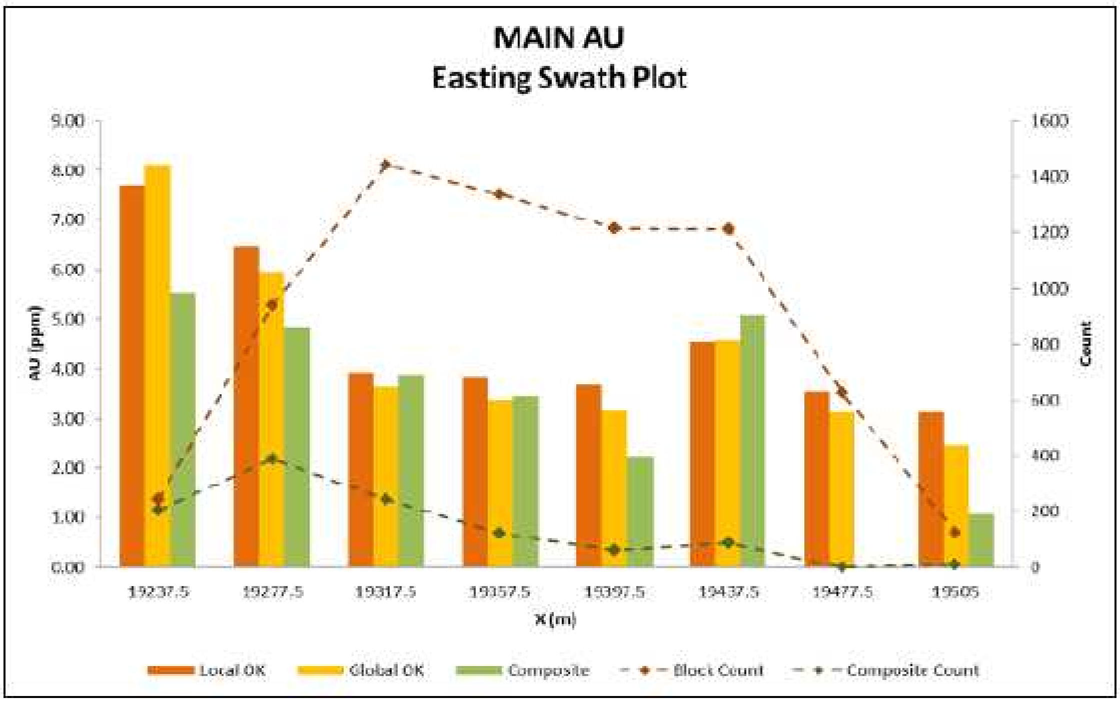


Figure 14-22: MAIN – Au Global OK – Swath plots

The gold estimations were then compared against the theoretical block distribution defined by a change of support calculation on the Gaussian anamorphosis of the sample distribution. Figure 14-23 compares the estimates with the theoretical block distribution for gold in the MAIN domain. The Local OK displays a sharp tonnage and grade gradient between gold grade cut-off of 3g/t and 4g/t. The grade tonnage curve for Global OK shows a better correlation with the theoretical block distribution. The metal tends to be consistently under-estimated compared to the theoretical block distribution. The theoretical block distribution is considered to predict tonnage, grade and metal closer to the expected global recovery. Therefore the Global OK is considered a better estimate at this stage of the mining project. All further references to the gold estimate in this report refer to the Global OK Au estimate.

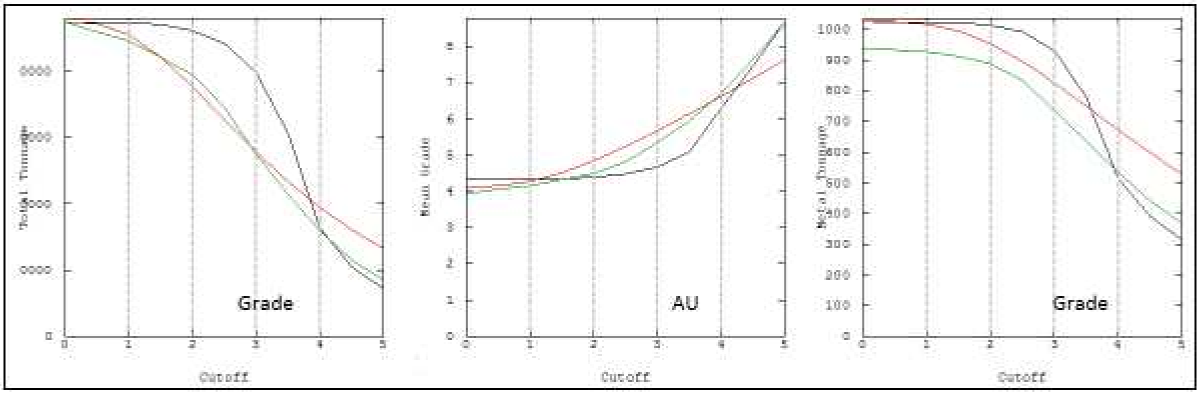


Figure 14-23: MAIN－gold－ Grade tonnage comparison Reasonable Prospects of economic extraction

From Left to right: Tonnage, Grade, Metal, Green = Global OK, Black= Local OK and Red= Theoretical Block distribution In assessing the criteria for reasonable prospects of economic extraction both open pit and underground scenarios were considered. With respect the scattered lower grade mineralization contained within the eastern domains (0p1E and 0p75E) near surface a simple pit optimisation using the optimistic parameters in Table 14-24 at twice the current gold spot price did not generate a pit of practical size on the eastern domains. All material in the eastern domains is not considered to have reasonable prospects of economic extraction and does not appear in the Mineral Resource.

Table 14-24: Pit optimisation parameters for evaluation of Mineral Resource classification

Parameter	Value
Gold Price (twice current spot)	AUD2,830 /Oz
Processing Cost (Whittle)	AUD58.4 /t mill feed
Recovery Oxide and Transition	81%
Recovery Fresh	90%
Overall wall slope angles	34 degrees

With respect to the underground potential the grade is reasonably consistent down to approximately 650 mRL below which it drops significantly. All material below 650 mRL is not considered to have reasonable prospects of economic extraction and does not appear in the Resource.

14.18 Classification

Classification is based on a combination of drill spacing, geological interpretation confidence, proximity to the previously mined open pit, reasonable prospects of economic extraction and grade. The classification areas are coherent zones and do not contain isolated blocks of lower classifications within them.

Measured is defined by the main, minor and 0p75W domains above 950 mRL and between plunging north and south boundaries that approximate the limits of the closer drill spacing (Figure 14-24).

Indicated (Figure 14-25) is defined by:

1. The 0p1 domain external to the Measured above 950 mRL; and
2. The approximate limits of the 20 x 20 m drilling.

The inferred is the remaining material above 650 mRL within approximately 50 m of drilling and the areas with low geological confidence in the orientations of the controls on mineralization and assumed structures (Figure 14-26).

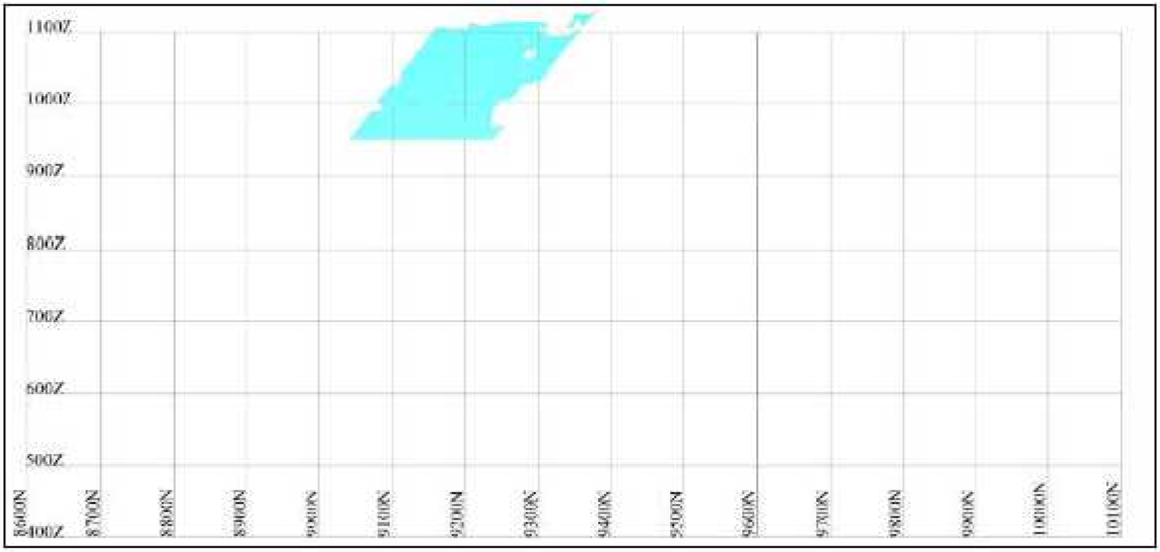


Figure 14-24: Long section facing west displaying measured blocks for all domains

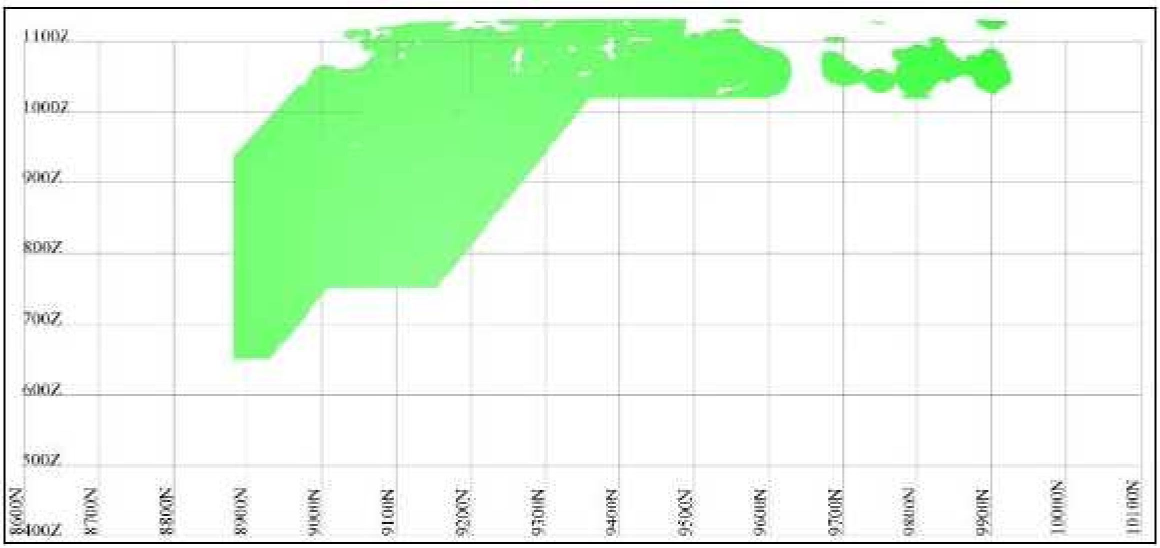


Figure 14-25: Long section facing west displaying indicated blocks for all domains

Figure 14-26: Long section facing west displaying inferred blocks for all domains

14.19 Mineral Resource Tonnage and Grade

Given the Mineral Resource is amenable to open cut and underground mining a split set of cut-offs is used for reporting. An elevation limit of 950 mRL has been used for the depth limit for open cut reporting as this is 50 m below the simplistic whittle pit optimisation depths generated with optimistic revenues.

A cut-off of 0.5 g/t Au is defined as the base case (Table 14-25), a comparison at 1.0 g/t Au is also included (Table 14-26). The open pit Mineral Resource is exclusive of the underground Mineral Resource. The Mineral Resources are stated here for the Maud Creek deposit with an effective date of 15 March 2016

Table 14-25: Open pit Mineral Resource above 950 mRL at 0.5 g/t Au cut-off – base case

Classification	Tonnage (kt)	Grade (Au g/t)	Contained Metal (KOz Au)
Measured	1,070	5.6	190
Indicated	1,100	2.1	75
Measured and Indicated	**2,170**	**3.84**	**268**
Inferred	530	1.4	25

It should be pointed out the Mineral Resource estimate is categorized as Measured, Indicated and Inferred as defined by the CIM guidelines for resource reporting. Mineral resources do not demonstrate economic viability, and there is no certainty that these Mineral Resources will be converted into mineable reserves once economic considerations are applied. The Measured, Indicated and Inferred Mineral Resource estimate has been prepared in compliance with the standards of NI 43-101 by Danny Kentwell, FAusIMM.

Notes to Table 14-15:

1. CIM definitions followed for classification of Measured, Indicated, and Inferred Mineral Resources.
2. Mineral Resources estimated as of 15 March 2016.
3. Mineral Resources stated according to CIM guidelines.
4. Totals may appear different from the sum of their components due to rounding.
5. Reported at a 0.5 g/t cut-off grade.
6. The open pit Mineral Resource is exclusive of the underground Mineral Resource.
7. The Mineral Resource estimation was performed by Danny Kentwell FAusIMM fulltime employee of SRK Consulting, who is a Qualified Person under NI 43-101.

Table 14-26: Open Pit Mineral Resource above 950 mRL at 1.0 g/t Au cut-off – for comparison only

Classification	Tonnage (kt)	Grade (Au g/t)	Contained Metal (koz Au)
Measured	1 067	5.59	192
Indicated	1 100	2.14	76
Measured and Indicated	**2167**	**3.84**	**268**
Inferred	232	2.36	18

The underground Mineral Resource consists only of material below 950 mRL. The base case is stated at 1.5 g/t Au cut-off (Table 14-27). A comparison at 2.0 g/t Au cut-off is provided in Table 14-28. The underground Mineral Resource is exclusive of the open pit Mineral Resource.

Table 14-27: Underground Mineral Resource below 950 mRL at 1.5 g/t Au cut-off – base case

Mineral Resource Classification	Tonnage (kt)	Grade (Au g/t)	Contained Metal (koz Au)
Measured	-	-	-
Indicated	4,330	3.28	456
Measured and Indicated	**4,330**	**3.28**	**456**
Inferred	1,450	2.65	124

It should be pointed out the Mineral Resource estimate is categorized as Measured, Indicated and Inferred as defined by the CIM guidelines for resource reporting. Mineral resources do not demonstrate economic viability, and there is no certainty that these Mineral Resources will be converted into mineable reserves once economic considerations are applied. The Measured, Indicated and Inferred Mineral Resource estimate has been prepared in compliance with the standards of NI 43-101 by Danny Kentwell, FAusIMM.

Notes to Table 14-27 and 14-28:

1. CIM definitions followed for classification of Measured, Indicated, and Inferred Mineral Resources.
2. Mineral Resources estimated as of 15 March 2016.
3. Mineral Resources stated according to CIM guidelines.
4. Totals may appear different from the sum of their components due to rounding.
5. Reported at a 1.5 g/t cut-off grade.
6. The underground Mineral Resource is exclusive of the open pit Mineral Resource.
7. The Mineral Resource estimation was performed by Danny Kentwell FAusIMM fulltime employee of SRK Consulting, who is a Qualified Person under NI 43- 101.

Table 14-28: Underground Mineral Resource below 950 mRL at 2.0 g/t Au cut-off – for comparison only

Mineral Resource Classification	Tonnage (kt)	Grade (Au g/t)	Contained Metal (KOz Au)
Measured	-	-	-
Indicated	3,490	3.65	410
Measured and Indicated	**3,490**	**3.65**	**410**
Inferred	1,026	3.04	100

It should be pointed out the Mineral Resource estimate is categorized as Measured, Indicated and Inferred as defined by the CIM guidelines for resource reporting. Mineral resources do not demonstrate economic viability, and there is no certainty that these Mineral Resources will be converted into mineable reserves once economic considerations are applied. The Measured, Indicated and Inferred Mineral Resource estimate has been prepared in compliance with the standards of NI 43-101 by Danny Kentwell, FAusIMM.

15 Mineral Reserve Estimate

Mineral Reserves are not being reported.

16 Mining Methods

The open pit design work carried out by SRK was based on the preferential allocation of potential mill feed to the proposed underground operations and oxide mineralization being treated as waste. With the proposed treatment of mineralization at the Union Reefs Processing Plant, the Oxide or contained within the pit design was report and included in the proposed mill feed.

Considering oxide as income generating provides the potential to increase the size of the open pit. The impact of the project economics are potentially large, particularly if the constraints of the waste dumps or underground preferences are relaxed.

Should the project proceed SRK recommends that a range of sensitivities/ scenarios are considered at a high level to inform the project team of the implications of these constraints. The sensitivities should include combinations of considering the open pit oxide mineralization as a mill feed source with both the underground preferences retained and removed. Conventional open pit mining techniques are proposed.

The underground is designed using conventional bottom-up open stoping mining methods utilising a combination of cemented rock fill and waste fill. The stopes will be accessed via a decline and level intervals and stoping lengths based on the geotechnical guidelines discussed below.

At this stage of the study, the design includes decline access from a portal located on surface outside the footprint of the open pit. The opportunity exists to modify the design to interact with the open pit design but this will potentially have constraints on production continuity.

16.1 Geotechnical

In conjunction with the Mineral Resource review, a comprehensive review was undertaken of the available data and analysis to determine the open pit and underground geotechnical design guidelines.

An assessment of overall slope angles and underground mining parameters has been undertaken by SRK using geological and geotechnical drilling data supplied by Newmarket Gold. The analysis provides good early-stage design guidelines of the geotechnical properties of the rock mass. The typical geotechnical conditions on site can be summarised as follows:

- The Hanging wall Tuffs are typically massive but may be locally bedded. Hanging wall tuffs are also affected by the numerous shears present in the Hanging wall, resulting in reduced strength, increased fracture frequency and graphitic and/ or chloritic alteration of the rock mass.
- The Footwall Sediments consist of low to medium strength thinly bedded or laminated mudstone and siltstone, and medium to thickly bedded sandstone. Zones of intense shearing with chlorite and graphite alteration occurring in the 5 to 10 m below the mineralized zone where the sediments are commonly black, highly graphitic and/or chloritic, very weak and fissile.
- The competency of the mineralized zone can be expected to be variable with competent, partially silicified mineralized zones separated by zones of intensely sheared rock.
- The distribution of the various fault configurations is not understood at this stage and this should be one of the main focus for subsequent field investigations.

16.1.1 Level of Confidence

The perceived level of confidence in the different data streams available were rated subjectively, as shown in Table 16-1. A five-point rating scale of Very Low – Low – Moderate – High – Very High has been used.

At this stage of the project development, at least a Low confidence rating should be expected for all items, with all items requiring further investigation. Aspects for which no data is currently available or represent a key concern have been flagged as Very Low.

Table 16-1: Qualitative risk assessment of study components

Data	Confidence level
Empirical: rock mass characterisation	Low
Structural: major structures	Low to Moderate
Structural; rock mass structures	Low
Rock mass strengths	Very low
Rock material strengths	Very low
Groundwater conditions	Very low
Slope angle recommendations	Low

16.1.2 Project Risks and Opportunities

The main project risks relate to the absence of data required for the development of design parameters. The areas where data is required are:

- Minor geological structures – additional pit mapping/photogrammetry and diamond drilling with geotechnical and structural logging are necessary to overcome the current orientation bias and increase the data density throughout the study area.
- Rock mass characterisation and strength – further drilling and evaluation is needed to improve confidence in the rock mass properties of all the domains, in particular within the major fault zones.
- Rock material strength – a representative laboratory testing program needs to form part of the next stage of investigation. This should allow the identification and evaluation of design mechanical properties which can be used for numerical analysis of pit designs at the next stage of the study.
- Improved understanding of the project groundwater conditions is needed to understand the interaction between the site hydrogeology and the pit walls.

16.1.3 Review of Geotechnical data

Geotechnical data was collected from drillholes drilled between 1990 and 2011. The geotechnical data was collected from the deposit and the rock mass adjacent to the deposit, with the more recent drilling collecting data throughout the wider area. Recent drilling was also more likely to include a wider range of geotechnical parameters. Sediments located in the footwall of the deposit are not well represented within the overall geotechnical dataset. Additional geotechnical drillhole data will be required for future studies. Geotechnical laboratory testing will be required from samples recovered from future drilling.

16.1.4 Rock mass characterisation

Geotechnical domains have been determined for both the open pit and underground areas at the Maud Creek project based on geological units, weathering, structural setting and rock mass quality.

Figure 16-1 shows the different geotechnical domains selected for the open pit analysis. For underground component of the study, stope stability assessments considered domains within the deposit and 10 m either side of the deposit only, whilst broader domains were used to characterise the rock mass for ground support requirements.

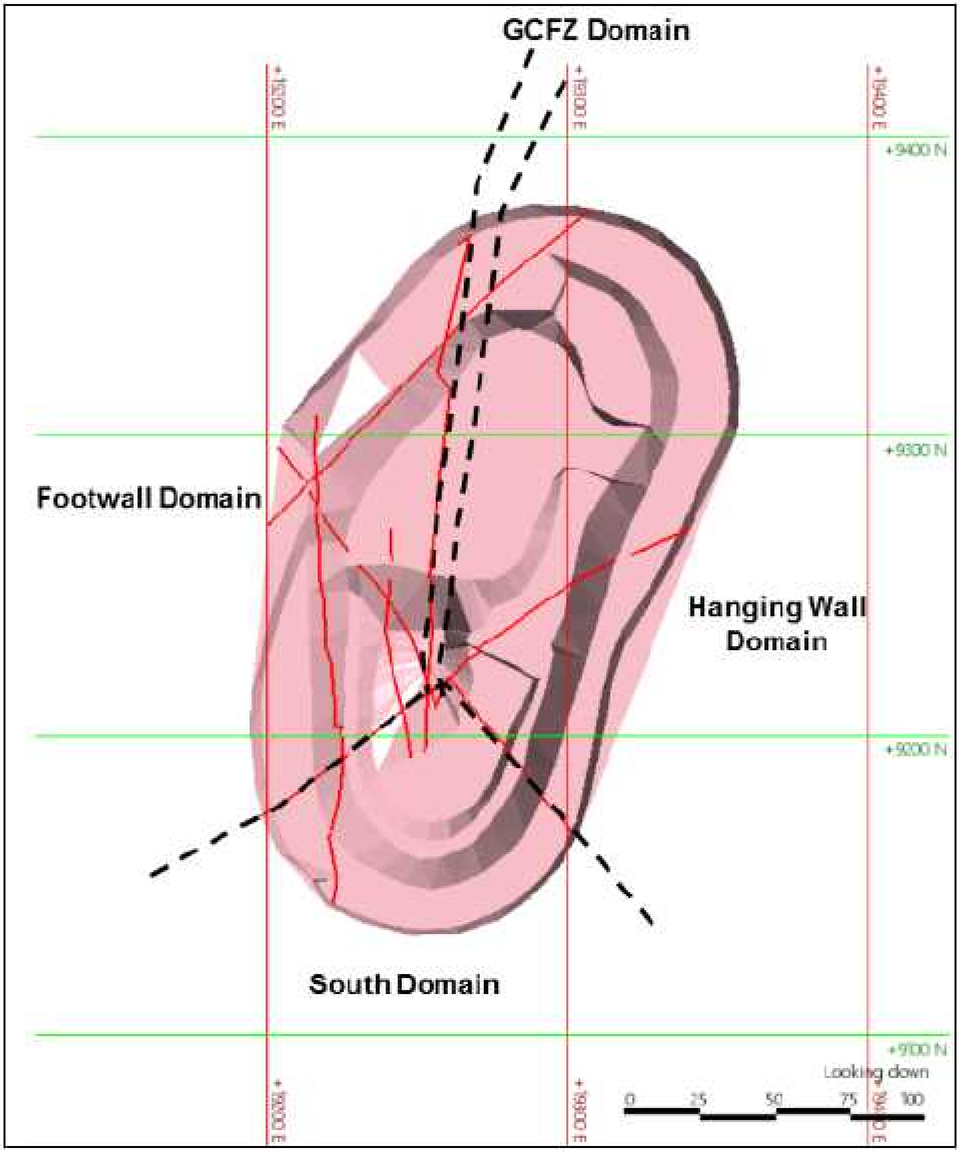


Figure 16-1: Plan view of existing pit showing the geotechnical domains used for rock mass characterisation

16.2 Underground Design Considerations

Newmarket Gold requested that SRK Consulting (Australasia) Pty Ltd (SRK) investigate the potential for processing mineralization from the Maud Creek deposit at the existing Union Reefs facility to eliminate the capital cost of constructing a process plant at Maud Creek. Offsite processing would likely result in tailings becoming unavailable and therefore paste fill no longer being a viable backfilling material. Modifications to the backfilling strategy, mining method and sequence will form part of the investigation.

16.2.1 Mining Method

The proposed mining direction is a bottom up sequence retreating to a central access and utilising waste rock fill with cemented rock fill in strategic locations. The mining direction requires the separation of the deposit into multiple mining areas to provide earlier mill feed.

The establishment of three mining areas has been considered and would require the creation of two artificial sill pillars at the base of each mining area to allow the later extraction of the pillar separating the areas. The artificial sill pillars are proposed to be constructed by placing cemented rock fill (CRF) as backfill in the stopes on the lowest level in each mining area. It is recommended that the bottom level of the mine be filled with CRF to allow mining at depth at a later stage.

16.2.2 Stope Stability

The change of mining method and modification of mining sequence will not change the stable stope dimensions. However the changes may require the final stopes in each level have longer strike lengths for a more practical final extraction. The longer strike lengths would be approximately 20-25m. The longer strike lengths would affect stope stability in footwall sediments, and also exceed the strike length design recommendations of 10m and 15m for footwall sediment domains A and B respectively (Table 16-2) (SRK 2015).

Stope stability and overbreak for the larger central stopes can be managed by designing stable beams between the stope footwall and shear zones or by installing cablebolts in the footwall or hanging wall. The equivalent linear overbreak slough (ELOS) for the increased strike lengths would be up to 2.0m for an unsupported stope in the footwall sediments domains A and B. ELOS for other geotechnical domains remains unchanged.

Stope dilution may be controlled by designing stable beams between the stope footwall and the shear zones, reduction in stope strike spans or cable bolt reinforcement of stope hanging wall or footwall.

Table 16-2: Stope Geometry

Domain	N'	Hydraulic Radius (HR)		Strike length (m)	ELOS (m)	ELOS 25 m strike (m)	Stope Height (m)
		Unsupported	Supported				
Sediments FW- A	0.9 - 3.27	2.3 - 3.8	6.5 - 8.0	10	1.0	2.0	25
Sediments FW - B	1.83 - 18.58	3.0 - 7.1	7.2 - 11.3	15	1.0	2.0	25
Sediments FW- C	0.6 - 30.39	2.0 - 8.4	6.1 - 12.5	20 - 30	1.0	1.0 - 2.0	25
ORE	1.66 - 26.89	2.9 - 8.1	7.0 - 12.1	15	-	-	10 - 35
Tuff - HgW	9.82 - 79.93	5.5 - 12.0	8.0 - 15.0	-	0.5	0.5	25

16.2.3 Stope Ground Support Requirements

The development ground support requirements were assessed using empirical methods and are summarised in Table 16-3.

Table 16-3: Ground Support Requirements

Geotechnical Domain	Excavation Type	Minimum Ground Support Requirements (Range)
Footwall Sediments	Main Decline	2.4 m Galvanised Friction bolts (46 mm diameter Split Sets) Bolt spacing 1.0 m - 1.6 m Surface support ranges from Fibrecrete (75 mm) to Mesh covering backs and walls to grade line
Footwall Sediments	Level Accesses	2.4 m Galvanised Friction bolts (46 mm diameter Split Sets) Bolt spacing 1.0 m - 1.6 m Surface support ranges from Fibrecrete (75 mm) to Mesh covering backs and walls to grade line
Vein (Deposit)	Ore Development	2.4m Galvanised Friction bolts (46 mm diameter Split Sets) Bolt Spacing 1.4 m - 1.8 m Mesh support covering backs extending to grade line
Hanging wall Tuff Development	Other Capital Development	2.4 m Galvanised Friction bolts (46 mm diameter Split Sets) Bolt Spacing 1.5 m Fibrecrete (40 mm) to Mesh covering backs and walls to grade line

Crown Pillar assessment

Caving and potential subsidence was identified as a possible hazard during this study. An empirical assessment was completed to determine the stability of the crown pillar located between the top of the stoping and the base of the proposed open pit.

A number of mitigation measures are available and include limiting the number of stopes open at a given time and limiting stope widths to 15 m with a 10 m thick crown pillar. Alternately increasing the crown pillar thickness would allow for wider stopes in the transverse stoping area. Cemented backfill is recommended for the stoping area located immediately below the open pit and highly weathered zones.

Further investigation of the crown pillar coupled with numerical modelling would allow optimisation of the crown pillar thickness.

16.2.4 Stress

Stress measurements have not been completed for the Maud Creek project. A low to medium stress environment has been assumed for this study based on the depth below ground surface and expected regional stress regime. It is not expected that stress related instability will result from the proposed mining strategy. It is highly recommended that samples recovered from future drilling be sent away for Acoustic Emission (AE) stress measurement testing to confirm the stress field. Stress measurement testing should be followed by numerical stress modelling.

16.2.5 Backfill

Waste backfill is considered suitable for most areas within Maud Creek, where the mining method allows and sequence is bottom up. Cemented Rock Fill (CRF) will be required for strategic areas and sill pillar extractions as described above.

SRK experience at similar mines that utilise CRF backfill for sill pillar extractions suggest that a mix design of 7% cement (by weight) for stopes up to 10m wide is a reasonable assumption for this level of study. CRF with 4-5% cement has been used to create stable vertical exposures in preference to leaving mineralized pillars unmined. A detailed CRF mix design should be completed in future studies.

16.2.6 Further work

Recommendations regarding phase two work, summarised in Table 16-3, includes the following:

Dedicated geotechnical drilling program to:

- Target areas that require additional data such as the footwalls sediments, and crown pillar area;
- Drilling proposed locations of LOM infrastructure – portals, declines, vent shafts etc.;
- Improve understanding of structural characteristics including continuity and orientation variability;
- To collect representative samples for laboratory testing;

Mapping and photogrammetry in current pit;
Geotechnical laboratory testing using NATA certified laboratories;
Detailed backfill design for potential mining methods (including CRF);
Potential stress measurement testing (AE or DRA); and
Numerical modelling of proposed mining layout and sequence (including crown pillar and central pillar sequence)

Table 16-4: Drill program targets and estimated depths

Description	Drillholes (No.)	Depth (m)	Total (m)
To investigate sediments at depth	4	600	2,400
To supplement data and sediments across mid depths of mine	4	500	2,000
Portal and boxcut investigation drill holes	3	50	150
To investigate Infrastructure located in sediments (can be separated into multiple shorter drill holes)	1	600	600
Open Pit geotechnical holes – will provide data on crown pillar area and upper sediments	6	150	900
Total meters for proposed program			6,050

16.2.7 Open pit Slope Stability Analysis

The preliminary open pit design recommendations have been based upon the geotechnical domains summarised in Table 16-4.

Table 16-5: Preliminary Slope Design Parameters

Domain	Geotechnical Safety Berm Width (m)	Ramp Width (m)	Bench Height Cases (m)	Depth Range (m)	Bench Height (m)	Bench Face Angle	Bench Stack Height (m)	Bench Stack Angle	Inter- Ramp Angle	Limit Over Slope A
Hanging Wall	N/A	24	All	0 - 25	12.5	55°	25	46°	38°	
			12.5	25 - 50	12.5	55°	25	46°	38°	
				50 - 100		60°	50	45°	33°	34°
				50 - 150			75	44°	35°	36°
				50 - 250					36°	36°
			25	25 - 50	25	55°	25	55°	46°	
				50 - 100		60°	50	54°	41°	43°
				50 - 150			75	53°	40°	42°
				50 - 250					42°	43°

Domain	Geotechnical Safety Berm Width (m)	Ramp Width (m)	Bench Height Cases (m)	Depth Range (m)	Bench Height (m)	Bench Face Angle	Bench Stack Height (m)	Bench Stack Angle	Inter-Ramp Angle	Limit Over Slope A
Southern	15	24	All	0 - 25	12.5	55°	25	46°	38°	
			12.5	25 - 50	12.5	55°	25	46°	38°	
				50 - 100			50	42°	34°	35°
				50 - 150			75	41°	35°	36°
				50 - 250					34°	35°
			25	25 - 50	25	55°	25	55°	46°	
				50 - 100			50	50°	38°	39°
				50 - 150			75	48°	40°	41°
				50 - 250					39°	40°
Footwall and GCFZ	15	N/A	All	0 - 25	12.5	55°	25	46°	38°	
			12.5	25 - 50	12.5	55°	25	46°	38°	
				50 - 100			50	42°	35°	33°
				50 - 150			75	41°	36°	35°
				50 - 250						34°
			25	25 - 50	25	55°	25	55°	46°	
				50 - 100			50	50°	40°	37°
				50 - 150			75	48°	41°	40°
				50 - 250						39°

16.3 Open Pit Mining

Open pit and underground mine planning was conceptually undertaken to estimate the mining inventory of the project. The mine plan includes Inferred Resources and scoping study level assumptions supporting the modifying factors and design guidelines, hence the mining inventories from the mine planning do not meet should Mineral Reserves as defined by the NI 43-101 instrument.

16.3.1 Open Pit Optimisation

Open pit optimisation was used to identify the optimum economic pit shape based on the highest project cashflow. The pit optimisation process seeks a solution to a complex 3D mathematical relationship involving the Mineral Resource model, geotechnical slope guidelines, product revenue, project constraints, modifying factors and costs.

The key inputs into the optimisation process include:

- Overall pit slope angles based on geotechnical recommendations;
- Mining costs estimates;
- Mining dilution and mining recovery parameter estimates;
- Material re-handling and processing costs;
- Processing recoveries;
- Product price and revenues;
- Selling and transportation costs (including rail and port costs); and
- Royalties and corporate overheads.

The Mineral Resource model was converted to a mining model by a process of regularisation to account for a selective mining unit size of 10 m x 10 m x 5 m vertical. Mining dilution of 10% and production losses of an additional 10% were applied in the optimisation and downstream mine planning processes to account for the anticipated selective mining.

The inputs used in the optimisation process are presented in Table 16-1.

Table 16-6: Optimisation parameters

Description	Optimisation Input
Software Used	Whittle™ Optimisation Software
Pit Geotechnical Design Parameters	Overall slope of 37 degrees
Mining Dilution and Ore- loss Philosophy	Mining recovery 90% Mining dilution 10%
Mining costs	AUD4.25/t mined. Applied as a single cost to all benches. Cost is supported by provided client information
Processing costs	AUD58.4/t transitional and fresh including allowances for G&A and concentrate transport and handling. Based on 600kt/pa Processing Plant,
Gold Price	AUD1,450/oz
Processing recovery	95% Fresh 85% Transitional Oxide not considered as economic
Product transportation cost	Concentrate handling costs included in processing costs.
Resource categories considered	All mineralization including Measured, Indicated and Inferred as defined in the Mineral Resource model.
Underground interface	Underground interface sensitivity analysis considered 95% mining recovery and only considers fresh mineralization. Underground mining costs of AUD49.65/t applied.
Limitations Applied	A limit was placed on the open pit mining at 1015 mRL elevation to leave the crown pillar in lower grade material.

The refinement of the inputs was progressive, particularly around the underground to open pit interface. A maximum open pit mining depth was manually applied to the 1015 mRL elevation to limit the access of the open pit to the higher grade mineralization. This depth limitation ultimately had limited impact on the final pit shell selection as the non-elevation restricted pit shells did not extend significantly deeper.

The selected base case optimisation pit shell which considers the underground to open pit interface offered an inventory of 634 kt at a grade of 5.12 g/t Au containing 104 koz of gold.

A range of solutions were conducted to test the sensitivity of the project to changes in base assumptions. The project was found to be sensitive to the consideration of underground mining. The underground mining analysis assumed an underground mining cost of AUD49.65 /t and 95% underground mining recovery. Removing the underground option increased significantly increased the pit shell inventory to 600 kt at an average grade of 5.54 g/t Au containing 107 koz gold.

The optimisation considered no revenue recovery for the oxide mineralization. SRK notes that future consideration of processing oxide mineralization offers the potential for a reduced strip ratio and a larger open pit. Physical limitations with the interface between the underground mine may ultimately restrict the pit size and overarch a future opportunity to consider oxide processing.

The cut-off grades reported by the optimisation processes were 1.65 and 1.48 g/t Au for the Transitional mineralization and Fresh mineralization respectively.

The relatively high cut-off grades are due to the high total processing and transportation charges.

16.3.2 Mine Design

The base case optimisation pit shell was used to guide the open pit mine design. The design parameters, presented in Table 16-2 were used in the mine design.

Table 16-7: Open Pit Design Parameters

Description	Design Input
Site layout	Brown fields site with prior open pit workings
Pit Geotechnical Design Parameters	37 degrees used in the optimisation
Minimum Mining Width	10 m x 10 m
Internal pit ramp parameters	18 m double lane ramp, 9 m single lane
Staged design logic	Top-down development with no internal staging
Waste Dump parameters	18 m wide ramp, 20 degree batter slope, 10 m wide berm, 15 degree overall slope

The geotechnical mine design recommendations recommended overall pit slopes of between 34 and 38 degrees depending on the pit sector. The mine design utilised 55 degree batter slopes, 13.5 m wide berms, on a 25 m high bench to achieve a 45 degree inter-ramp angle. The measured slopes range between 30 and 38 degrees depending on the pit sector measured and the number of ramp intersections in the sector. As the open pit design is shallow and small, the ramp has a large impact on flattening the pit slopes.

The pit ramps were positioned to exit to the west of the operation in an alignment very similar to the existing as-built pit. This access aligns with the waste dump and RoM pad location concept layouts.

SRK lengthened the pit ramp in an effort to recover more of the targeted mineralization. In doing so, SRK utilised the existing operations ramp on the west and northern side of the existing pit to align the new pit ramp. This incurred waste material outside the pit shell but allowed the pit depth to increase. The resulting design depth of the pit was the 1078 mRL level compared to the base case pit shell reaching 1060 mRL. The net result is the open pit design has a mining inventory of 168 kt compared to the base case optimisation pit shell with 634 kt of mineralization.

16.3.3 Mine Production Scheduling

An open pit mining schedule was developed to demonstrate the open pit mine production over time. The pit is has an inventory of less than a year' s mill feed. The criteria used for the open pit mining schedule included:

- In situ Mineral Resource model used, scheduling all confident classifications of mineralization including measured, indicated and inferred resource categories;
- Cut-off grades applied to RoM mineralization;
- Oxide mineralization treated as waste;
- Quarterly periods;
- The mill production rate 600 ktpa and assumed constant;
- Top-down schedule with overburden mining in Quarter 1; and
- Total material movement targeting 1,000 kt per quarter

The open pit mining schedule was based on scheduling in situ materials using cut-off grades and the conversion of in situ mineralization to potential mill feed undertaken by including mining loss and dilution post scheduling.

Oxide mineralization was flagged in the production schedule to enable it to be independently reported.

Vertical rate of advance was not used as a scheduling constraint. The resultant schedule is aggressive with the 50 m deep pit being mined-out in 7 quarters. This is considered achievable if operational delays for grade control and pit dewatering are actively managed.

Salient points from the schedule include:

- Open pit mine life of 7 Quarters;
- Quarter 1, used for removing overburden and a cutting back around the existing pit;
- Quarter 2, commencement of production; and
- Quarter 3-7, lower strip ratio with the majority of the open pit mining in this period.

Table 16-8 shows the breakdown of the material movements by quarterly period. The reported information is based on in situ mineralization and subject to cut-off grades. All oxide mineralization is included in the waste totals in this report.

Table 16-8: In situ material scheduling

	Mineralization above cut-off grade		**Waste**
Period	**Inventory (‘000 t)**	**Grade (g/t Au)**	**Inventory (‘000 t)**
Quarter 1	0	0.0	900
Quarter 2	101	4.0	900
Quarter 3	143	5.2	800
Quarter 4	100	5.4	800
Quarter 5	100	5.4	800
Quarter 6	100	5.4	800
Quarter 7	89	5.4	0
Grand Total	**634**	**5.1**	**5,000**

A high level summary of the RoM production schedule is shown in Figure 16-3. Oxide mineralization has been reported separated. Transitional and fresh RoM mineralization has been subject to cut-off grades and mining loss and dilution estimation.

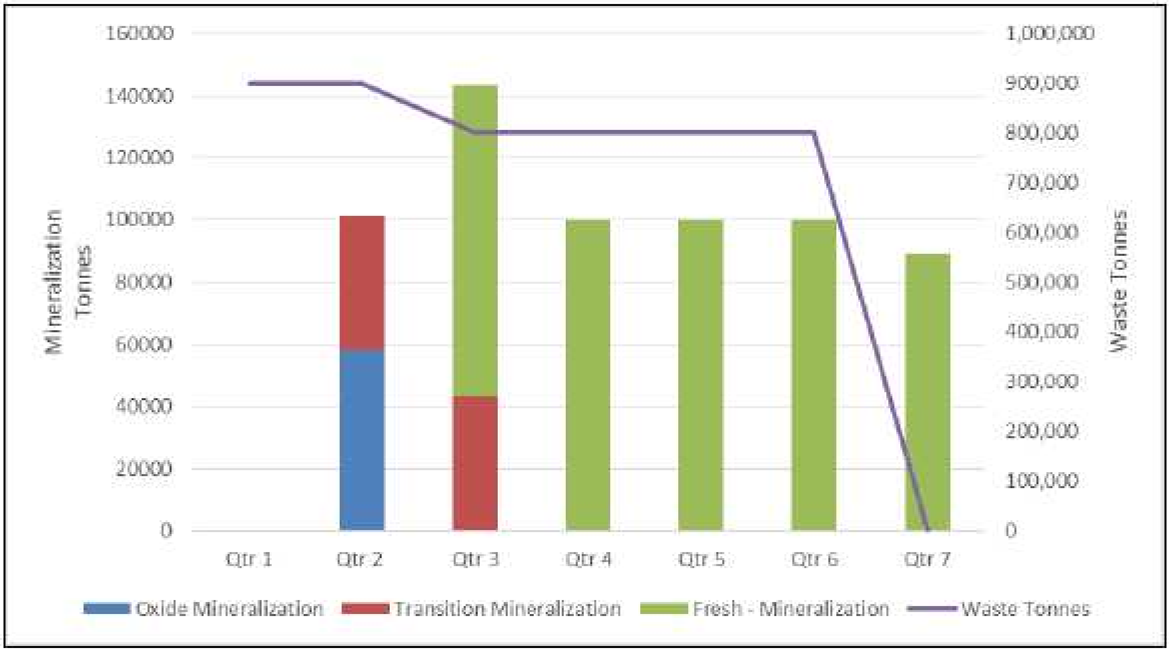


Figure 16-2: Total material movement

16.3.4 Waste and Stockpile Design

A waste dump was designed to the west of the open pit. The dump access ramp was aligned to minimize the haulage distance of waste from the pit to the waste dump. The design parameters and assumptions used for the waste dump are:

- 18 m wide double lane ramp;
- 20 degree batter slope;
- 10 m wide berms on 10 m high benches;
- 15 degree overall slope;
- Maximum total height of 20 m; and
- Standoff of 85 m from the waste dump to the pit crest.

The waste dump size was set by the following inputs and assumptions:

- The mine plan indicates open pit waste production of 5.0 Mt.
- In situ density of the open pit waste estimated at 2.4 t/m^3.
- The underground mine produces 722 kt of waste.
- Total waste production is estimated at 5.7 Mt.
- Assuming an average 30% swell factor for the truck compaction in the dumps, the loose density of the waste is estimated at 1.85 t/m^3.
- The surface water diversion works require 200,000 m ^{3} of waste which equates to 370,000 t.
- Underground mining requires the equivalent of 3 Mt of in situ waste, the balance of waste material is will be stowed in a waste dump.

16.3.5 Open Pit Mining Method

The open pit mining operation proposed is short term with a short operating life of 9 months. The mining equipment considered is small scale. A conventional 80 t class excavator would be ideal as the loading unit and can be matched to 85 t class rigid frame mine trucks for haulage. The length of haul is anticipated to vary from a 2 to 3 truck haul.

The productive mining fleet is anticipated to be supported by a combination of a water truck, grader and bulldozer. This support fleet will maintain the haul road, pit floor, waste dump and drill and blast pattern preparation. Other minor equipment such as IT loaders, support trucks and explosives trucks will support the drill and blast and mobile equipment maintenance activities.

The bench configuration is anticipated to be 5 m drilling benches, mined in 2 x 2.5 m flitches, with the material types being defined by mark-out tape and paint as designated by the site geologists.

The grade control solution has not been defined at this stage of study but could use reverse circulation ahead of mining, blast hole sampling, and/or mapping and/or chip and channel sampling. Reverse circulation to full pit depth supported by mapping would be the preferred approach as this allows the bench plans to be rapidly developed and minimising grade control lost time to support the proposed fast vertical rate of advance.

Drill and blast is envisaged, recommend drill and blast to ensure productivity as free dig estimates are often optimistic. Drilling would be conducted by conventional top hammer rig with a single pass 5.0 m benches with 0.5 m sub drill. Wall control can be achieved using batter holes. Blasting is anticipated to primarily be performed using ANFO due to the dry conditions.

Pit dewatering is proposed to be managed using sumps and pontoon mounted pumps. Sumps will be progressively developed ahead of the bench mining to ensure dry mining conditions. The pontoon pumps will be used as required to dewater the sumps with the discharge water delivered to settlement dams prior to discharge.

16.3.6 Forward works

The open pit mine planning has been evaluated using a well-established, valid and appropriate approach. There are, however, a number of key assumptions in which the assessment is sensitive to and SRK recommends that these areas be subject to further definition as the study advances. These are outlined below:

- The open pit to underground interface and associated crown pillar has not been optimised. The open pit is currently giving preference to the underground mining operations by using relatively high mining recoveries in the interface section of the deposit. Future mine planning and optimisation of this interface is likely to increase the project value going forward.
- The open pit scale is sensitive to operating cost changes and further development of the project definition and cost estimation will help define a more robust open pit mine planning solution.
- Waste dump size. A consideration by SRK was minimisation of the project waste dumps need to be minimised. Larger open pits supported by revised future inputs and assumptions will generate larger waste dumps. Ideally a defined limitation on the size, footprint or otherwise could be stated and worked towards as a project target or limitation. .

16.4 Underground

16.4.1 Mine Access

Two options were initially considered for accessing the underground workings:

- Box cut from the surface; and
- Portal located in the pit.

Due to the size of the open pit and the scheduling requirement to access the underground early to ensure that there is production continuity, the in-pit portal option was not considered further.

The designed mine access is located to the north-east of the pit as shown in Figure 16-4.

Figure 16-3 and Figure 16-4 show the decline located in the footwall of the deposit and the decline is designed to a depth of, 500 m below surface, 650 mRL.

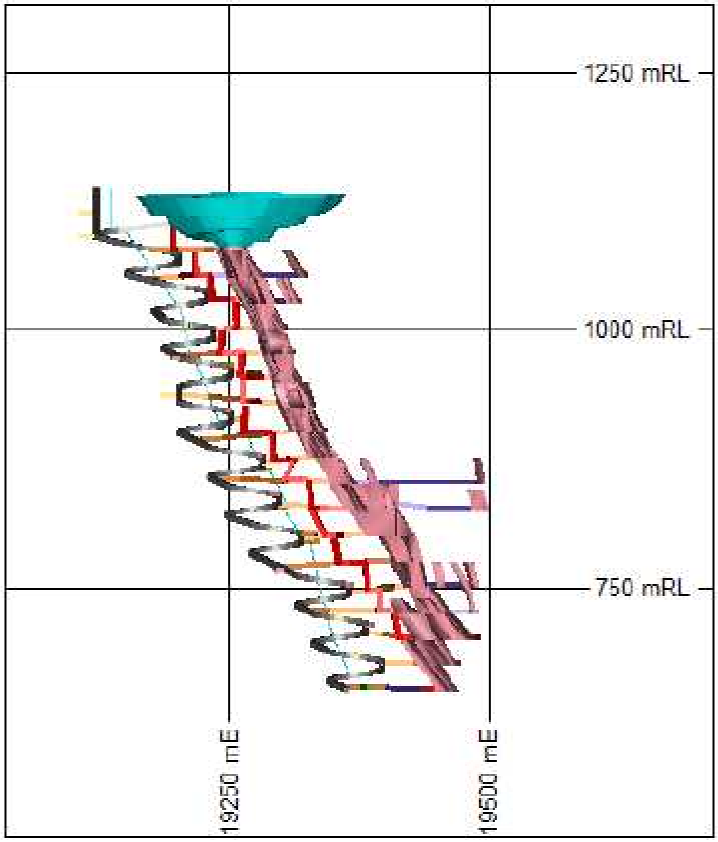


Figure 16-3: Decline Access – Section looking north

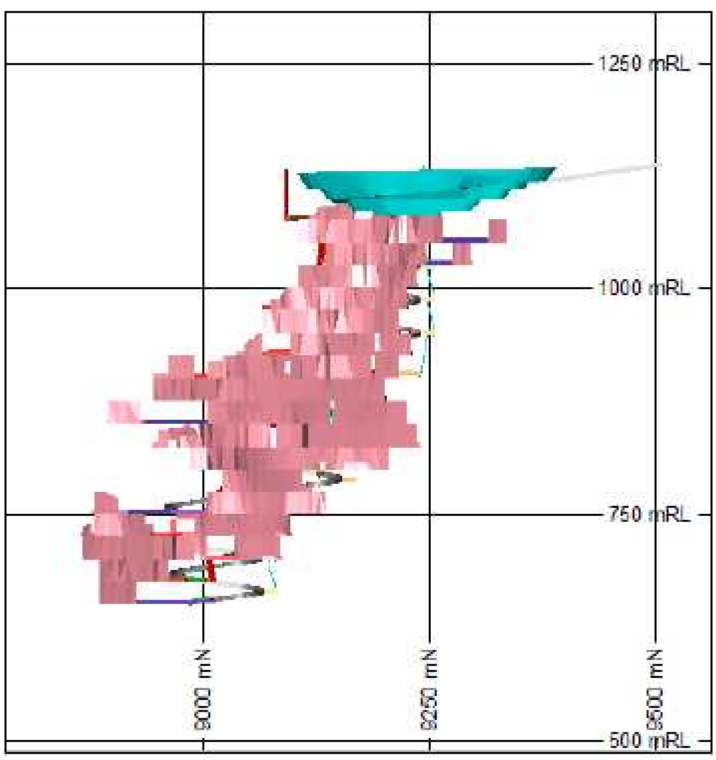


Figure 16-4: Long section looking west

16.4.2 Mining Methods

Underground mining methods can be categorized into three categories:

- Caving methods.
- Unsupported methods.
- Supported methods.

Given the type of mineralization and the width of the deposit caving methods are not appropriate for the project.

The unsupported and supported mining methods identified were appropriate to the deposit, based on the geotechnical review were:

- Long Hole Open Stoping; and
- Benching

These two methods are suitable given the deposit geometry and predicted ground conditions.

Further design work and cost analysis determined that Open Stoping is the preferred mining method for the deposit due to the lower development requirements.

16.4.2.1Mining Method Description

The proposed open stope mining method has sublevel spacing of 25 m and stope strike lengths between 10 m and 30 m depending on the ground conditions and ore continuity. The mining sequence Figure 16-5 retreats along the ore-drive towards the central access and is a bottom-up method. Cemented rock fill will be placed in the stopes as backfill to allow the mining of the next stope in the sequence. Two mining front on each level are possible.

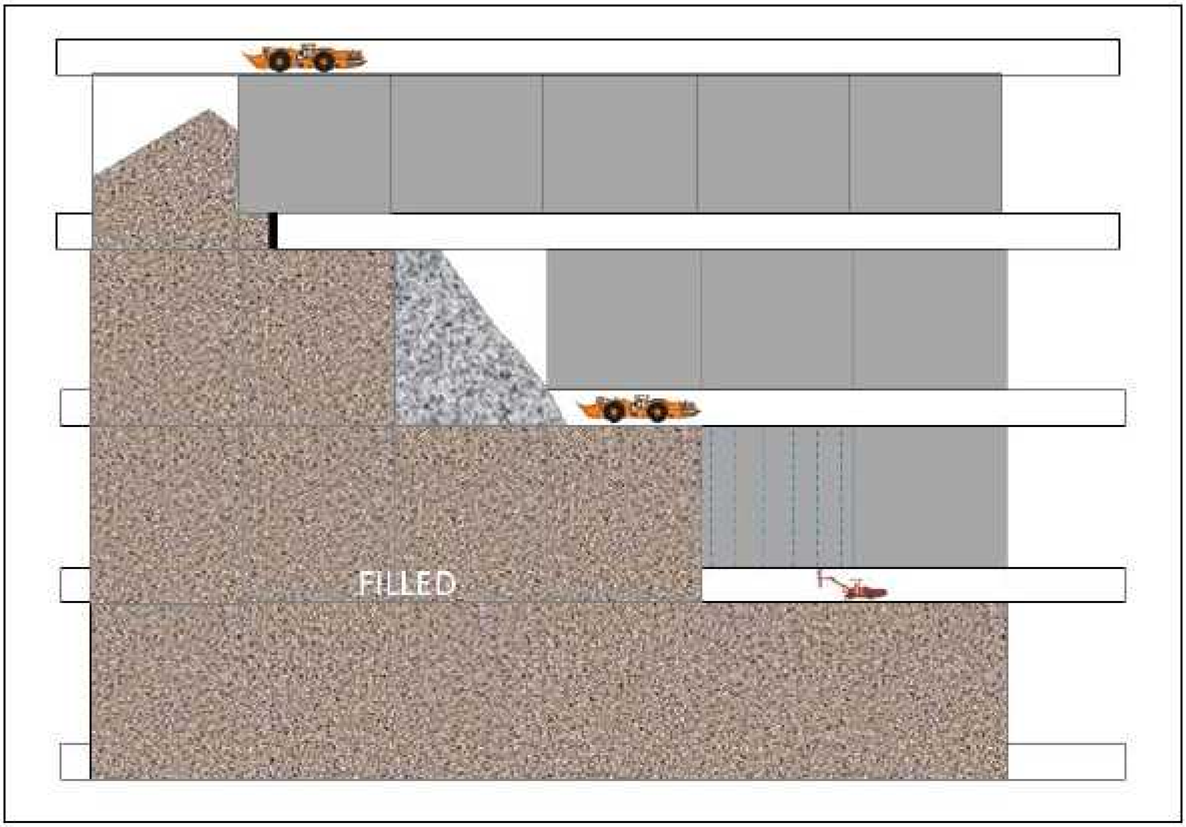


Figure 16-5: Long hole open stoping

16.4.3 Mine Design

16.4.4.1Cut-off Grade

Table 16-10 shows the cut-off grade inputs that have been applied for the Maud Creek underground design these resulted in a cut-off grade applied for the underground design of 2.65 g/t Au. This cut-off grade was applied as a "seed" value to identify the potential mining inventory.

Should the project progress, re-evaluation of the cut-off grade based on the findings of this study and future work is recommended.

Table 16-9: Cut-off Grade Inputs

Parameter	Unit	Value
Gold Recovery	%	92
Gold Concentrate Grade	g/t	60
Payable Gold	%	90
Gold Refining Charge	USD/oz	5
Gold Price	AUD/oz	1,415
Royalty	%	4
Exchange Rate	AUD: USD	0.85
Mining Cost	AUD/t	85.00
Processing Costs	AUD/t	25.00
General and Administration costs	AUD/t	15.00
Freight costs	AUD/oz	1.50

Figure 16-6 to Figure 16-8 show the sensitvity of the resource to the different design parameters that could be applied to the deposit at different cut-off grades.

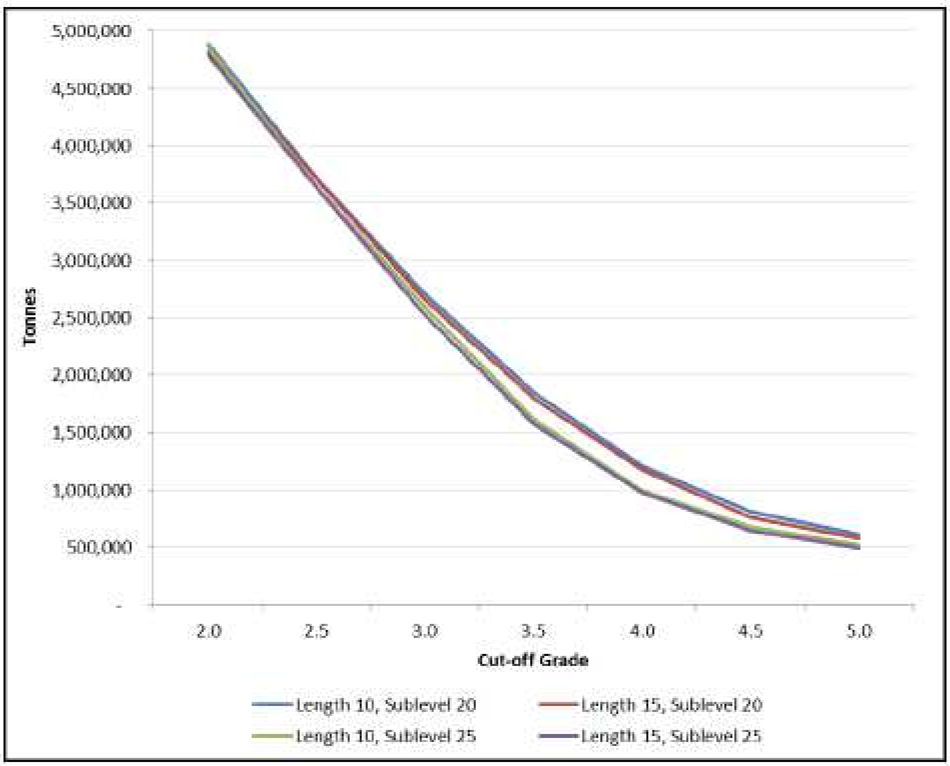


Figure 16-6: Cut-off Grade and Design Parameter Sensitivity – Tonnes

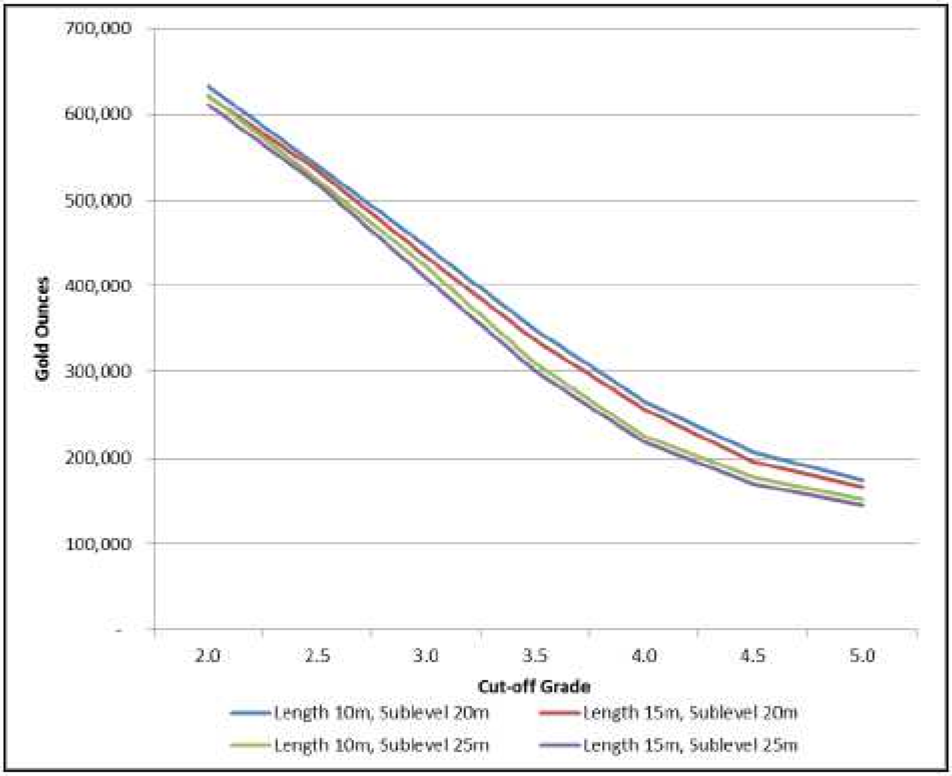


Figure 16-7: Cut-off Grade and Design Parameter Sensitivity - Ounces

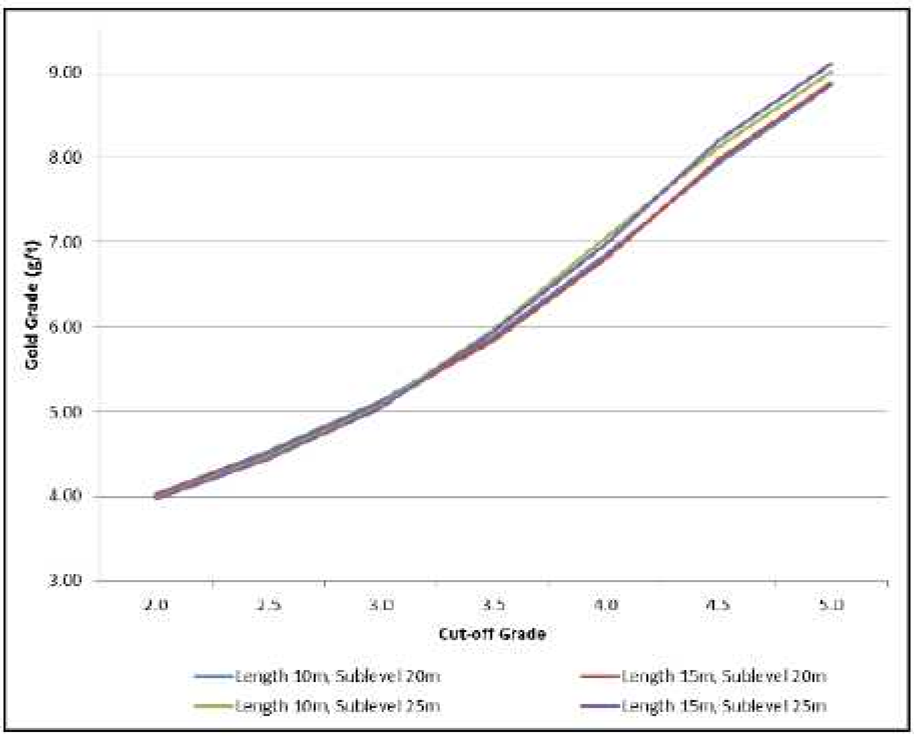


Figure 16-8: Cut-off Grade and Design Parameter Sensitivity - Grade

16.4.4.2Lateral Development

Each sublevel has an access drive, ventilation access and ore drives. The designed drive sizes are summarised in Table 16-11 and typical level presented in Figure 16-9.

Table 16-10: Lateral Development Design Parameters

Description	Size
Decline	5.5 mH x 5.5 mW
Stockpiles, Sumps	5.0 mH x 5.0 mW
Footwall Drive, Waste Drives	5.0 mH x 5.0 mW
Ventilation Access	5.0 mH x 5.0 mW
Ore Drives	5.0 mH x 5.0 mW

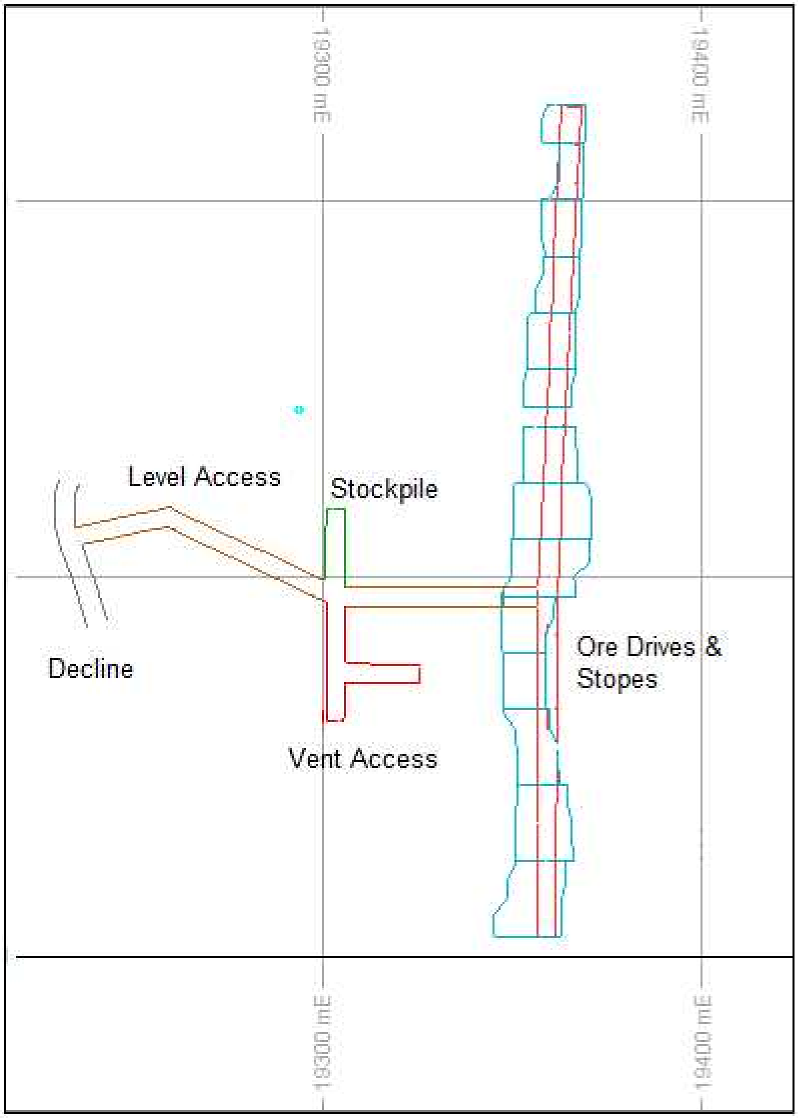


Figure 16-9: Typical sub level layout

16.4.4.3Vertical Development

The vertical development includes an escapeway and ventilation rises. Table 16-12presents the design parameters for the vertical development.

Table 16-11: Vertical Development Design Parameters

Description	Size
Return Air Rises	4.5 m x 4.5 m
Escapeway	3.0 m

16.4.4.4Ventilation Requirements

Fresh air will be drawn through the decline and escapeway rise and exhausted via the return rise system. The return air rise system is a series of 3 m x 3 m rises between levels to surface.

Total primary airflow requirement has been estimated based on 0.06m ^{3}/s for every kW power of mobile equipment according to Section 71.3 of the *Work Health and Safety (Mines) Regulation 2014* (NSW), which is the highest standard in Australia. The Northern Territory mining regulations do not provide any guidance. Table 16-13 shows the estimated peak production required airflow, and Table 16-14 presents the designed ventilation capacity of the mine.

Table 16-12: Required ventilation

Equipment	Power rating (kW)	No. of units	Total power (kW)	Total airflow (m^3/s)
Truck - 30t	293	3	879	53
LHD R1700	262	2	624	37
Service Vehicles	125	1	125	8
Light Vehicles	75	3	225	14
Total				**111**
Miscellaneous requirements (i.e. pump stations, explosive magazine, etc.)				**50**
System Losses (10%)				**16**
Mine ventilation requirements (with losses)				**177**

Table 16-13: Designed Intake and Return Air Capacity

Type	Description	Size	Capacity (m^3/s)
Intake	Decline	5.5 mH x 5.5 mW	182
	Escapeway	3.0 m	42.0
Total Intake			224
Friction Losses (10%)			22
Total intake capacity (with losses)			**202**
Exhaust	Return Air Rise	4.5 m x 4.5m	243
Friction Losses (10%)			24
Total exhaust capacity (with losses)			**219**

16.4.4.5Stoping

The stope design parameters are presented in Table 16-15. The parameters are based on the geotechnical domains and the cut-off grade from the preliminary cost estimates for the Project.

Table 16-14: Stope Design Parameters

Item	Value
Sublevel Spacing	25 m
Stope Length	10 - 30 m depending on ground conditions
Stope Width	4 -15 m
Cut-off grade	2.65 g/t

16.4.4.6Mine Service and Infrastructure

Water

Mine water will be supplied underground via the decline and service holes.

Dewatering of the underground mine will consist of a series of pump stations link via service holes, utilising the decline where required.

Compressed Air

Compressed air will be supplied underground from a surface compressor via the decline to each sublevel.

Power

The underground mine will require a 1000V power supply. The power will be distributed underground via the decline and service holes where required. The installation of underground substation has been allowed for in the design.

Emergency Egress

An emergency egress ladder way system is included in the mine design. Coupled with the decline, personnel will have a second means of egress from all sublevels within the mine.

During initial development it is recommended that refuge chambers be advanced along with the decline development face. Decline stockpiles can be converted to fixed refuge chambers as necessary to enable all personnel to be within close proximity of a refuge chamber or fresh air source.

16.4.4 Underground Dilution

There have been two types of dilution included in the Maud Creek design, planned and unplanned dilution. Figure 16-11 shows the definition of the dilution included in the Maud Creek mining inventory.

Planned dilution is the non-ore material (below cut-off grade) that lies within the designed stope boundaries (mining line). Unplanned dilution is additional material, which is derived from rock or backfill outside the designed stope boundaries. Unplanned dilution is predominately due to blast overbreak and sloughing of unstable walls. Unplanned open stope dilution is a measure of stope instability. Planned dilution can be controlled by optimizing the mining method and mining design.

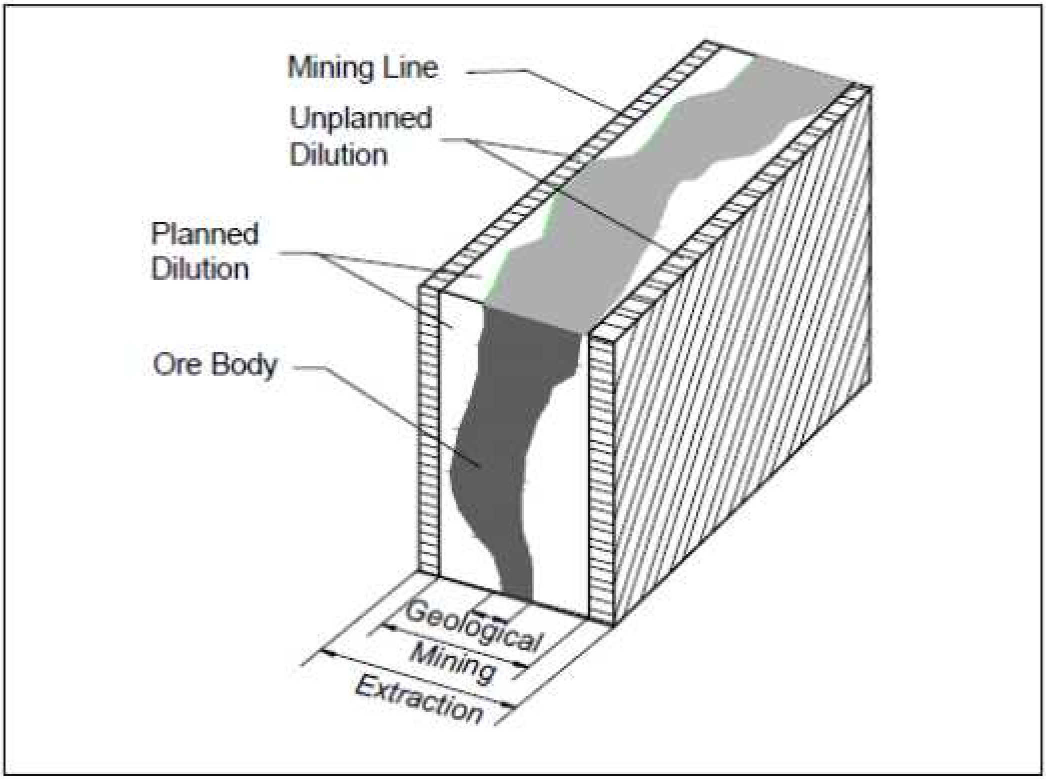


Figure 16-10: Dilution Definition
Source: A**fter Scoble & Moss, 1994)**

16.5.2.1Planned Dilution

The planned dilution is the dilution that is within the stope boundaries (mining lines, Figure 16-10). The range of the planned dilution for the mine design is summarised in Table 16-16. Where the planned dilution is greater than 15%, the stopes have been bulked out to the minimum mining width of 3 m.

Table 16-15: Range of planned dilution in mining inventory

Description	**Value**
Minimum	0%
Maximum	50.42%
Average	1.5%

16.5.2.2Unplanned Dilution

Unplanned dilution is the dilution that occurs beyond the stope boundaries (Figure 16-10). Typically, the factors that influence the unplanned dilution are shown in Figure 16-11.

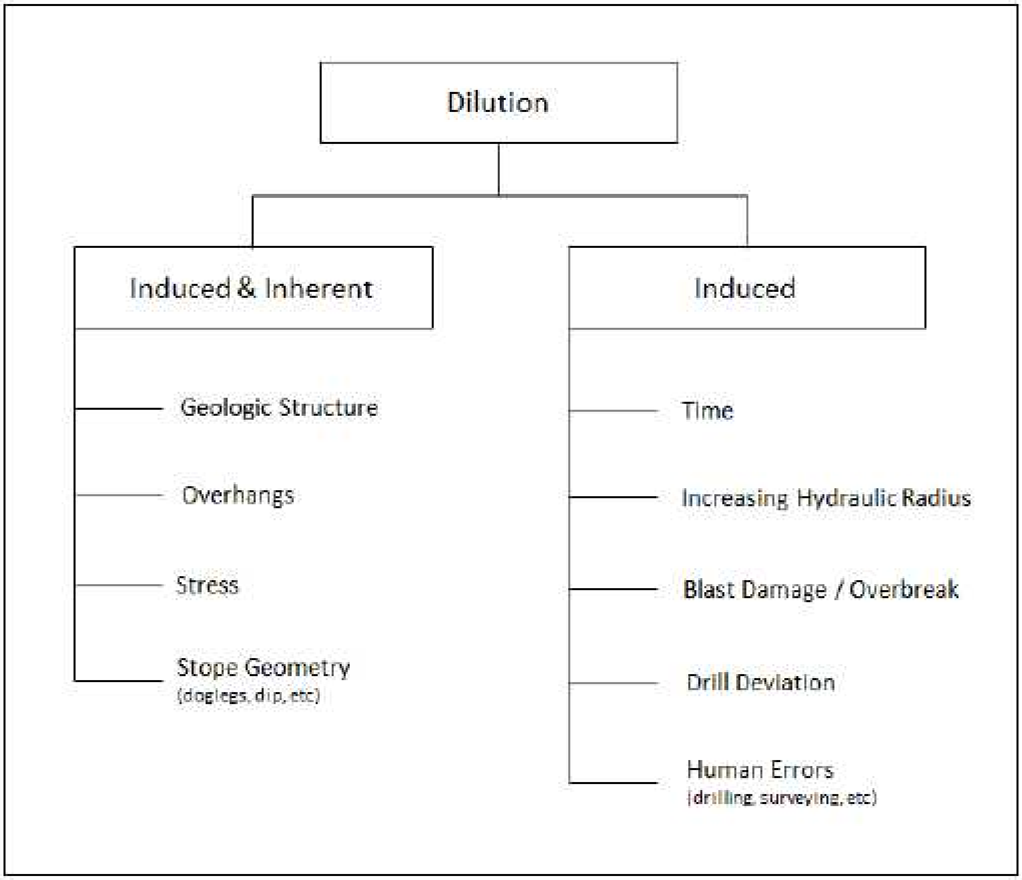


Figure 16-11: Influences on unplanned dilution

Source: After Pakalnis et al, 1995

The geotechnical assessment estimated that the equivalent linear overbreak slough (ELOS) for the hanging wall would be 0.5 m and the footwall 1.0 m. Using these figures the average dilution, 7% on the Hanging wall and 14% on the footwall has been applied to all stopes using a grade of 1 g/t Au.

16.4.5 Mining Recovery

The mining recovery has been calculated for the stope geometries designed, the average recovery has been calculated to be 95%. The calculations were based on the shape of the stopes and the expected material that will be unable to be loaded from the stopes. An angle of repose of 45 degrees has been assumed in these calculations.

Table 16-17 summarises the underground Mining Inventory for the Project by Resource classification, post application of dilution and mining recovery modifying factors. Figure 16-12 and Figure 16-13 present the tonnes and ounces by level.

Table 16-16: Underground Mining Inventory by Resource Classification

Resource Classification	% of Feed	Diluted Mined Tonnes and Grades		
		Inventory (kt)	Grade (g/t Au)	Contained Gold (koz)
Measured	20	679	6.1	132
Indicated	70	2,306	3.9	290
Inferred	10	291	2.8	26
Underground Mining Inventory	**100**	**3,276**	**4.0**	**423**

It is important to note that the PEA is preliminary in nature that it includes Inferred mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and there is certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have documented economic viability.

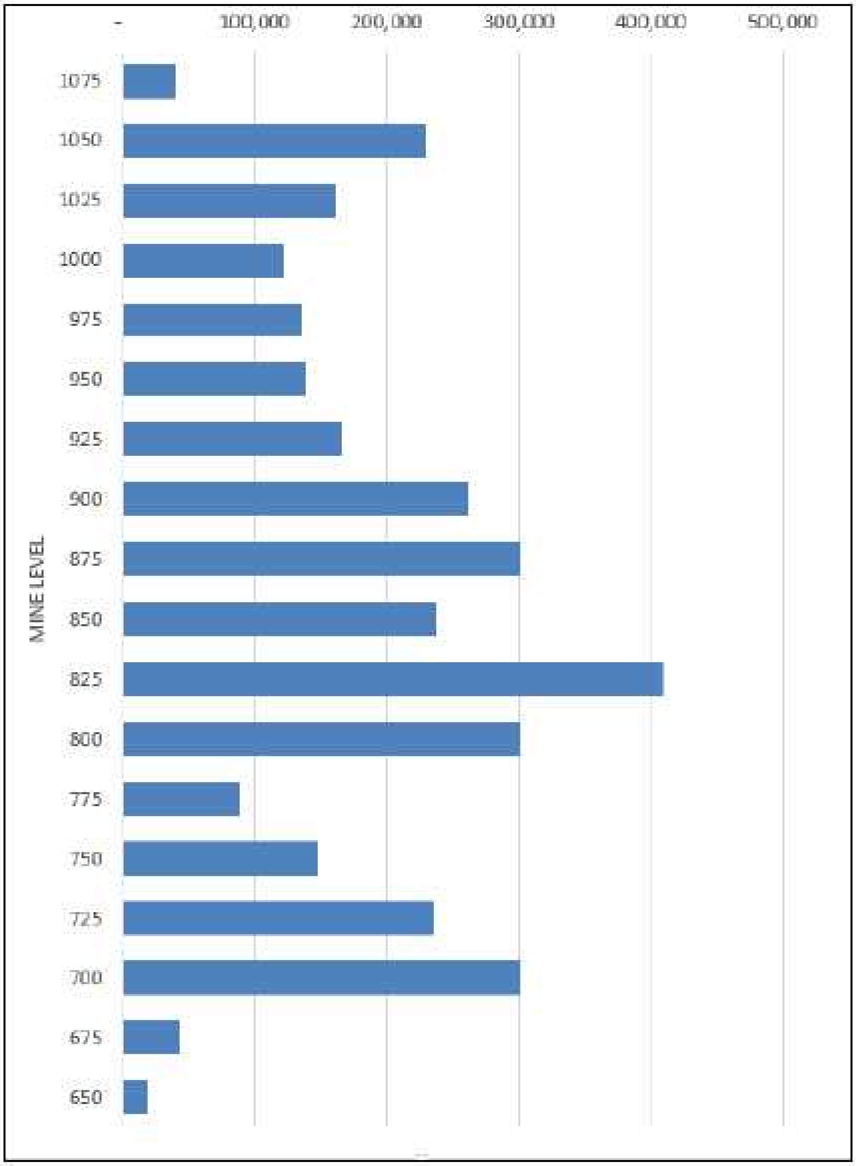


Figure 16-12: Chart showing production tonnes by Level

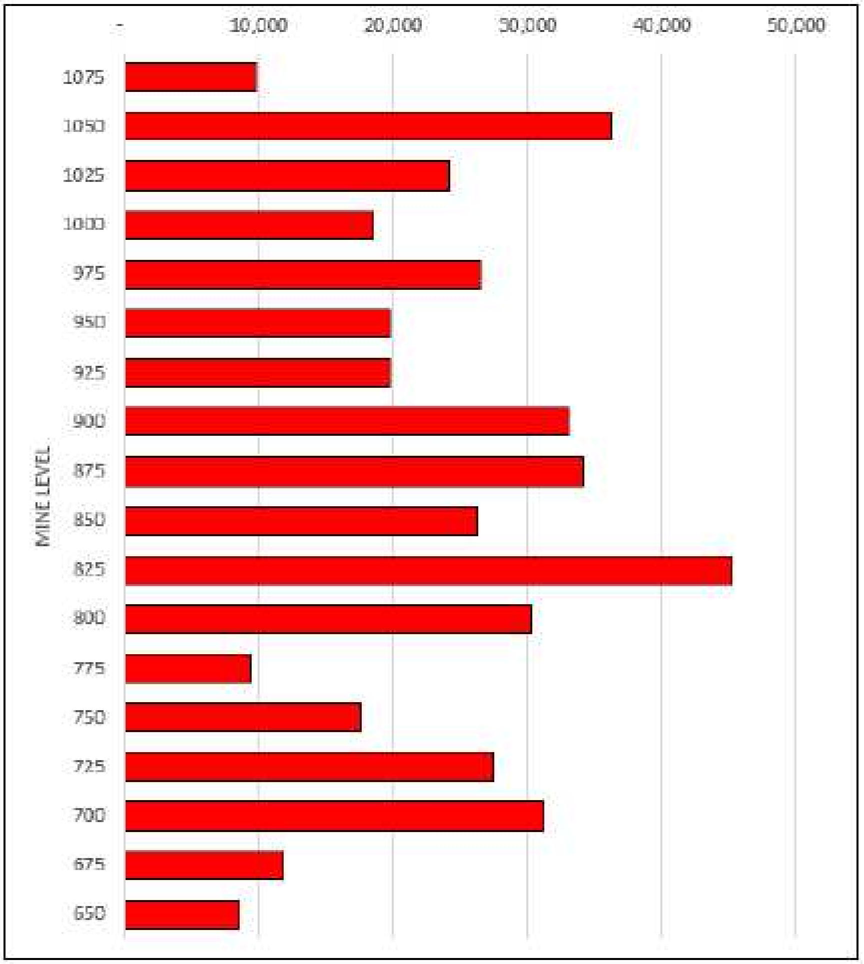


Figure 16-13: Chart Showing Ounces by Level

16.4.6 Underground Mine Schedule

The Maud Creek underground project schedule has the following key points:

- Mining activity is spread over a 6 year period;
- Steady state peak production target of 0.5 Mtpa ore production;
- First ore produced in Qtr 5;
- The production ramp-up period is 1.5 years, achieving peak production in Qtr 9;
- A maximum production rate of 0.5 Mtpa maintained for 5 years (average of 64 vertical m of production advance per annum);
- Simultaneous mining of multiple production fronts spanning multiple levels throughout the mine life will be crucial for maintaining a constant steady state ore production; and
- Production begins to ramp down after 4.5 years of full production because the number of active mining fronts begin to drop off as the known mining inventory is depleted.

Schedule Productivities

SRK has applied the productivities presented in Table 16-18 to the mining activities in the schedule.

Table 16-17: Typical productivities applied to the mining activities

Development	Units	Value
Single Heading		
Decline and other Capital Lateral Development	m/month	120
Operating Development	m/month	240
Shaft	m/day	3
Escape way	m/day	3
Rises	m/day	3
Multiple Heading		
Lateral Development capacity	m/month	60
Production Drilling	m/d	260
Stoping	t/d	840 (average per stope)
Backfill	m^3/d	1,900

Scheduling Strategy

The following scheduling strategy for Maud Creek deposit has been applied:

- Stope production to commence as soon as possible;
- A backfill crown pillar is created as shown in Figure 16- 14 to allow multiple advancing mining fronts at depth, this pillar is mined later in the schedule after the last stopes on the levels directly above and below have been filled;
- Development to be mined on a "just-in-time" basis;
- Return airways and escape way rises for each block to be completed prior to the commencement of stoping on each respective level;
- Scheduling of multiple production fronts over multiple levels are planned throughout the mine life are crucial for maintaining a constant steady state ore production.

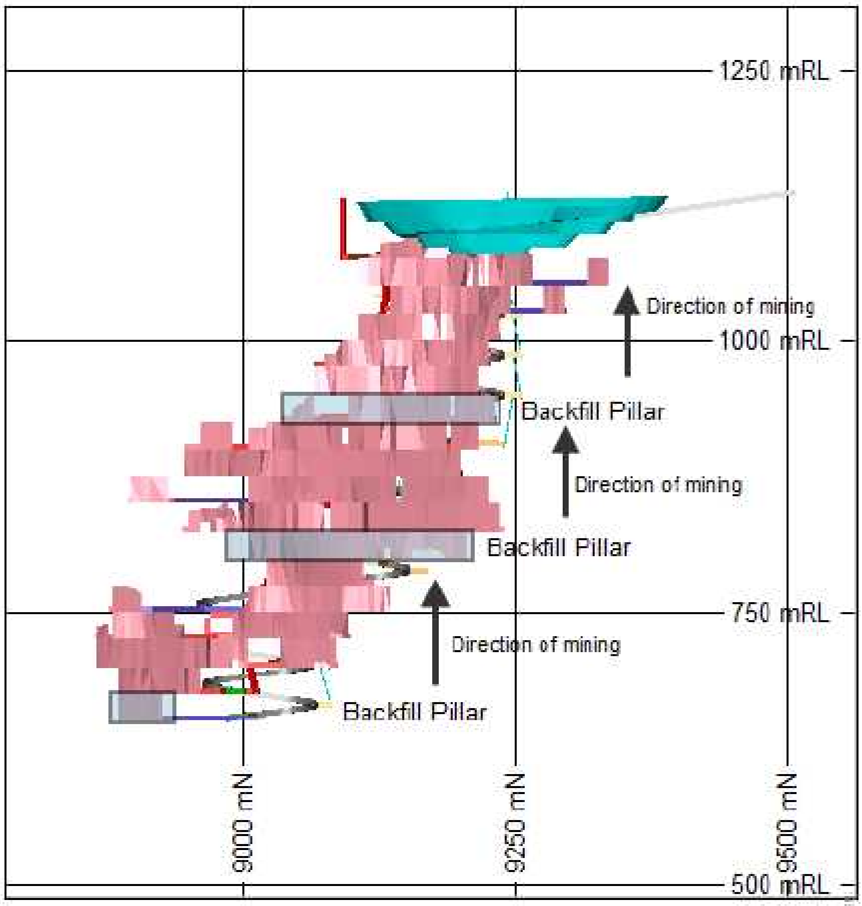


Figure 16-14: Long section looking west showing mining fronts

SRK applied Tatman' s (2001)[1] method to validate the suitability of the preliminary production rate for the Maud Creek deposit, shown in Table 16-18.

The average stope thickness varies between 4m and 35m in the transverse stopes. The average thickness is 10m. Using the table presented in Figure 16-15, SRK selected the deposit thickness row >10 m and the moderate risk column. The production rates resulting from the formula range from 0.25 Mtpa to 0.55 Mtpa. SRK selected a target production rate for scheduling of 0.5 Mtpa which falls within the moderate risk range and results in an average vertical m advance rate of approximately 64 vertical m per annum.

Table 16-18: Recommended rate multipliers by lode thickness

Deposit Thickness (m)	Low Risk	Moderate Risk	High Risk
>5 m	<20	20 to 50	>50
5 to 10 m	<50	50 to70	>70
>10m	<30	30 to 70	>70

[Annual production rate = Rate factor * Rate multiplier]

Rate factor = tonnes per vertical metre

Source: Tatman 2001

[1] Tatman, CA (2001). *Production-rate selection for steeply dipping tabular deposits,* Mining Engineering, *pp. 62-64,* October 2001.

Development Schedule

Figure 16-15 presents the quarterly target of approximately 450 capital development meters per quarter (104 per month) and 400 operating lateral development meters per quarter.

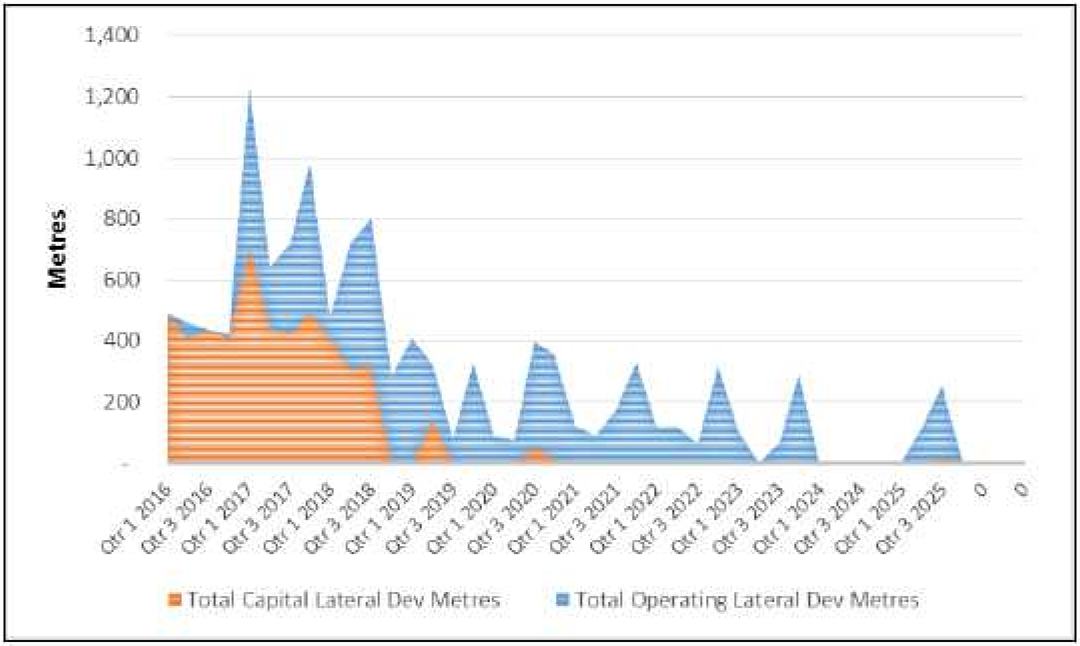


Figure 16-15: Development schedule

Underground Production Schedule

Figure 16-16 presents the underground production schedule and Figure 16-17 the ounces produced by quarter for the Project.

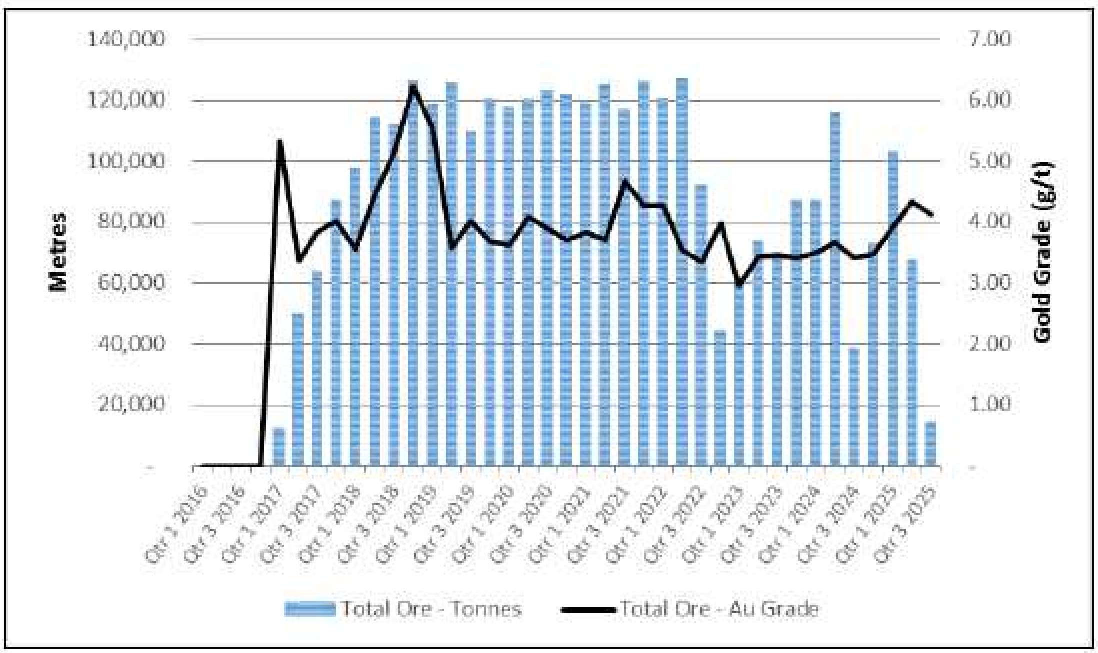


Figure 16-16: Proposed production tonnes and grade

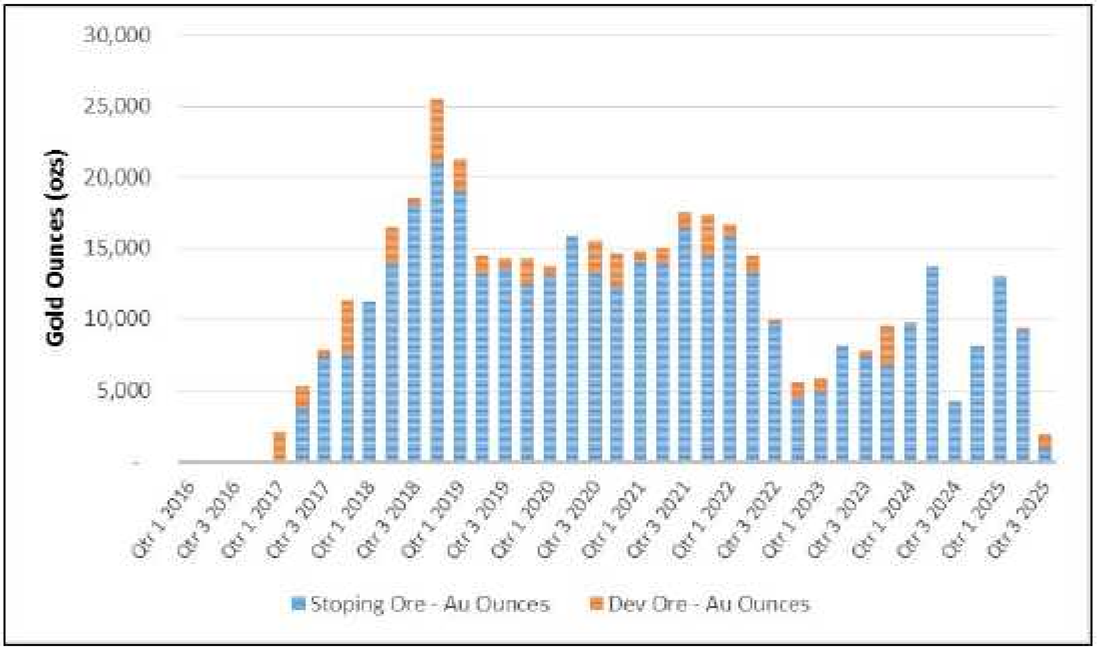


Figure 16-17: Gold Ounces by Quarter

Equipment Requirements

Table 16-19 summarises the equipment required to operate the underground. Allowance has been made for underground trucks, loader, drills and services vehicles.

Table 16-19: Equipment Requirements

Description	No. of Units
Trucks - 30t	3
Loaders -6 - 7 m^3 bucket	2
Development drills	2
Production drills	1
Cable bolter	1
Service vehicles	1
Light vehicles	3

Mobile Equipment

Table 16-20 lists the requirements for underground truck haulage; this is based on using 30 t underground haulage trucks. Table 16-21 lists the requirements for loaders, this is based on using 6-7 m^3 bucket loaders for both development and production requirements. Table 16-22 and Table 16-23 list the requirements for development and production drills.

Table 16-20: Trucks Requirements

Description	Units	Value
Peak time haul distance to surface	m	2,100
Average surface haul distance	m	400
Total travel time per cycle	min	24
Total loading time per cycle	min	5
Allowance for inefficiencies	min	5
Total cycle time	min	34

Description	Units	Value
Truck payload	t	30
Tonnes per hour per truck	t/hr	53
Average work hours per day	hr	18
Work days per year	day	365
Truck maintenance days per year	day	52
Effective work days per truck per year	day	313
Truck capacity per year	kt/year	298
Peak production (ore + waste)	kt/year	640
Required trucks at peak production	trucks	2.1
Total trucks required (rounded up)	**trucks**	**3**

Table 16-21: Loaders Requirements

Description	Units	Quantity
Loader production rate	t/day	1,000
Work days a year	day	360
Peak production rate (ore + waste)	kt/year	640
Required loaders	loaders	1.8
Total loaders required (rounded up)	**loaders**	**2**

Table 16-22: Required Development Drills

Description	Units	Quantity
Development drill rate		
Single heading	m/month	120
Multiple heading	m/month	240
Required development rate		
Single heading	m/month	104
Multiple heading	m/month	140
Required development drills		**2**

Table 16-23: Required Production Drills

Description	Units	Quantity
Drill density	t/m	12
Drill production rate	m/day	250
Work day per year	days	360
Peak stope production	kt/year	480
Required production drills	**drill**	**1**

Services Equipment

Table 16-24 lists the requirement for service vehicles, including service truck and light vehicles.

Table 16-24: Required Service Vehicles

Description	Number of units
Service Vehicles	1
Light Vehicles	3
Cabolter	1

16.5 Hydrogeology/ Dewatering

This section discusses the below-ground (Hydrogeology) aspects, the surface water management (Hydrology) aspects are discussed in Section 18.7.

SRK reviewed the findings and basis of previous reports and considers that them to be fit for purpose. Based on the absence of the underlying data, the purpose and stage of this study it was not appropriate to undertake additional hydrogeological modelling.

Should the study progress, additional data is required and should be incorporated with the revised structural / geological model to develop an updated numerical model. This would either build on the existing model, if data can be made available or a new model would be constructed. Specifically the updated model would confirm / address dewatering rate assumptions and would seek to collect data at depth. There is a distinct lack of knowledge at depth in the existing model.

16.5.1 Hydrogeology

Groundwater within the Maud Creek project area occurs within fractured, weathered and oxidised tuff, typically near the contact with main quartz vein. The quartz breccia zone located between the water table and around 100 mbs is interpreted to constitute a fractured rock aquifer. Groundwater may also occur at the contact and within the upper portion of the tuff (URS, 2007).

Groundwater recharge occurs by infiltration of rainfall. Flows in Gold Creek are also thought to be a possible source of groundwater recharge (URS, 2007).

The cavernous limestone of the Tindal Limestone aquifer to the west is likely to be hydraulically isolated from the fractured rocks and metasediments in the vicinity of the mine. Using test pumping data from two bores tested by Dames and Moore (1998), estimated values of transmissivity (T) ranged between 24 and 46 m^2/d. These values of T are considered relatively small and likely indicative of hard rock water bearing materials where fracturing is not well developed, poorly connected or both.

16.5.2 Dewatering

Previous Dewatering Analyses

Dames and Moore (1998) developed a groundwater flow model to determine the number of dewatering bores and pumping rates required to lower the water-table as part of planned open cut pit operation. The modelling found that seven dewatering bores with rates of between 300 and 500 m3/day (total dewatering rate of 2,200 m3/day) would be required to lower the water-table by 120m in an 18 month period (SRK, 2015).

URS (2007) expanded on the work of Dames and Moore (1998) by extending the model domain to simulate local and regional scale flow features. The model was used to predict dewatering of an existing open pit 200 m long by 100 m wide by 26 m deep and, in 2007, containing an estimated 300 ML of water (assumed mix of incident rainfall, minor runoff and groundwater inflow). The model extended to sufficient depth to assess the dewatering of underground mining to a depth of 700 m below surface (mbs) (-580 m RL). The model predicted dewatering rates of 3,400 m^3/day during the first year of mining, decreasing to 1,700 m^3/day after ten years of mining; this is similar to that proposed in this Technical Report.

Dewatering Assessment for Proposed Mine Development

Dewatering estimates for the proposed open pit and underground mine are based on the following assumptions and limitations:

The existing open pit excavation measures 200 m long by 100 m wide by 26 m deep and contains approximately 300 ML of water (mix of incident rainfall, minor runoff and shallow groundwater inflow).

The pit will need to be dewatered prior to underground mining.

The pit is within weathered and unweathered volcanics (tuffs).

The proposed open pit will be 300 m long, 200 m wide and 50 m deep and mined over a 9 month period.

Underground mining will commence from 50 mbs to about 500 mbs over a proposed 9 year period.

On-site hydraulic parameters are available from two pumping tests (in 1998) using bores which were drilled and completed to between 120 and 130 mbs and hydraulically connected to the various lithologies / aquifers over their entire depths. The reported parameters are considered reasonable for the lithologies cited given the paucity of actual field derived data.

There is no measured hydraulic data below 130 mbs. Noting the proposed mining development is expected to reach 500 mbs i.e. 370 m below the depth to which hydraulic parameters have been obtained.

Open Pit

Based on historical information and testwork, SRK estimated the groundwater inflows to the open pit to be 60 m^3/d (less than 1 L/s) based on the following relationships and assumed values:

$Q = \pi.K.ho^2 / \ln (ro/rpit)$

Where,

K = horizontal hydraulic conductivity (m/d) - 0.06 m/d

h_o = height of water surface above pit/aquifer base (m) - 25 m (assume only radial flow)

r_{pit} = radius of pit (m) - 80 m

r_o =radius of zero drawdown (m) - 530 m. In order to obtain ro the following relationship was used:

$r_o = (2.25.\ T.t/S)^{1/2}$

Where,

T = transmissivity (m2/d) - 35 m2/d

t = time since discharge commenced (day) - 90 days

S = storativity term (dimensionless) - 0.025

Dames and Moore (1998) assessed that to dewater a pit of roughly similar horizontal dimensions as proposed in this Technical Report, seven dewatering bores pumping at a combined rate of 2,200 m^3/d (about 25 L/s) would be required to lower the water table to 120 mbs over 18 months.

Given that there is no recent aquifer hydraulic data, scaling the Dames and Moore (1998) dewatering rates to suit development of a 50 m deep pit is suggested. On this basis and assuming a linear relationship between pit depth and required dewatering rate, a pit to 50 mbs would need to be pumped at about 900 m^3/d (approximately 10 L/s) over an 18 month period (ignoring the stored volume in the pit).

Underground Mine

URS (2007) developed a numerical groundwater model for a pit and underground mine, with the model extending to about 700 mbs. The potential underground mine is within the dimensions of the URS (2007) model.

The outcomes of the model were that in the first year of underground development, pumping rates would need to be of the order of 3,400 m^3/d (40 L/s) declining to 1,700 m^3/d (20 L/s) over a ten year mine life. Initial pumping rates could be reduced if dewatering commenced prior to decline development.

Summary

The dewatering assessment is deemed an appropriate starting point given the stage of project development and that no additional groundwater hydraulic data has been obtained since the Dames and Moore (1998) work.

There is considerable uncertainty in the dewatering assessment given the paucity of aquifer hydraulic parameters and that the data is only applicable to depth intervals of 120 - 130 mbs, whereas underground development is proposed to extend to about 500 mbs.

The following needs to be addressed to minimise the risk to the project from uncertainties in the dewatering assessment.

- The representivity of aquifer parameters derived from only two pumping tests given any structural complexity in and around the pit and underground mine;
- The anticipated depth of the underground mine to 500 mbs means that the lower 370 m of the proposed mine has no representative groundwater hydraulic data; and
- It is uncertain as to whether structural elements have been incorporated in the existing models. This is essential as groundwater storage and flow will largely be controlled by structure in this geological environment.

16.5.3 Recommended future Study Work

Field Investigations

To reduce cost, additional work especially drilling should be multi-purpose. For example, any additional resource estimation / sterilisation drilling could incorporate some or all of the following:

- Air circulation drilling: noting of lost circulation zones; monitoring of discharge rate over a v-notch weir; monitoring of water quality (physico-chemical) with depth.
- Fluid drilling (diamond etc.): noting of lost circulation zones; downhole geophysical logging; packer testing.

A sub-set of holes could be completed as groundwater level monitoring bores using small diameter PVC casing to provide greater spatial and vertical coverage of the site and regional groundwater systems. Several monitoring bores exist across the project area and these may be suitable as observation bores (e.g. for possible pumping tests) in-addition to the function they currently serve as part of the groundwater quality monitoring network. Several test dewatering bores were installed by Dames and Moore (1998), but the integrity and accessibility of these is unknown and should be assessed from bore condition surveys.

It is suggested that any deep holes be initially surveyed using Nuclear Magnetic Resonance (NMR) to provide downhole hydraulic data without the need for relatively expensive pumping or packer tests. The need for more detailed pumping or packer tests should be assessed based on the outcomes of the NMR surveys.

Installation of pressure transducer data-loggers in monitoring bores would be useful to assess relationships between rainfall and groundwater level trends as well as providing valuable data for hydraulic modelling calibration purposes.

Hydraulic Modelling

The following is suggested to improve the reliability of model predictions and thus minimise risks associated with inadequate dewatering knowledge.

- Incorporate updated geological modelling.
- Incorporate recent groundwater level monitoring data for improved calibration.
- Ensure structural complexity in the numerical groundwater model is sufficient to allow robust dewatering predictions.
- Incorporate any relevant downhole geophysical data acquired during field activities suitable for refining the numerical groundwater model.

The existing groundwater flow models should be used as the starting basis for any additional modelling, although this may require changes to the model geometry to reflect the current site and regional geological understanding. This assumes the existing groundwater flow models are readily accessible and importable into present versions of compatible software.

17 Recovery Methods

17.1 Background

Maud Creek sulphide (fresh) mineralization is refractory to direct cyanidation but respond well to flotation. There is significant but variable gravity-recoverable gold throughout. The gold is largely locked in sulphides and requires an oxidation process for downstream processing. Furthermore, the presence of preg-robbing carbonaceous material renders the mineralization 'double-refractory' .

The study and process design is focussed on the Maud Creek sulphide resource. However, the Maud Creek deposit also includes around 300 kt of oxide and 230 kt of transitional material. The oxide mineralization is generally amenable to direct cyanidation and will be treated through the existing Union Reefs Processing Plant. The transitional mineralization is less amenable to conventional cyanide leaching (is more variable) so alternatively, this transitional material may be treatable by flotation. Flotation recovery could be further improved following a sulphidization pre-treatment process. Controlled potential sulphidization (CPS) technology has been successfully applied on several gold and copper/gold mines, and may be beneficial at Maud Creek. Some preliminary testing would be worthwhile.

The main body of metallurgical testing and other development work was conducted from 1994 - 1998 for Kalmet Resources NL and Kilkenny Gold NL. Some minor work was done for Harmony Gold Operations Ltd in 2003, and then further testing and piloting were undertaken for Terra Gold Mining Ltd in 2006. Along the way there have been several engineering and consulting reports in support of the metallurgical development of the project. In particular, John W MacIntyre and Associates Pty Ltd supervised the earlier testing, starting in late 1996 and culminating in a detailed report: "Metallurgical Evaluation of the Maud Creek Project" in September 1998.

This review draws heavily on the MacIntyre report and the original metallurgical testwork reports, and also considers the later testwork where appropriate. The process design presented herein is based on the available information that is considered by SRK to be appropriate for this PEA. Some minor gaps remain and further development and optimisation work is recommended. Furthermore, the present mine plan is considerably different to that envisaged in the late 1990s so the original design recommendations must be treated with caution.

The process design philosophy is to add a simple, conventional, inexpensive and low-risk flotation circuit to the existing facilities at the Union Reefs Processing Plant, allowing flexibility with respect to circuit configuration and downstream processing. The base case presented here is to modify the Union Reefs Processing Plant to allow sulphide treatment (campaign treatment) by adding a flotation concentrator plant at the Union Reefs site, producing a gold sulphide concentrate for sale or possible in-house processing. The sale options are discussed briefly in Section 19, while the downstream treatment options considered in Section 17.7.

The Union Reefs Processing Plant currently treats free milling ores from the Cosmo underground mine by conventional crushing and grinding, gravity recovery and CIL processing. It has excess crushing capacity and two underutilised grinding mills at the current throughput rates on a hard underground mineralization. From 1999 to 2003, annual processing rates of 2.8 Mtpa were achieved, but it currently treats only ~750-850 ktpa, so it can handle the additional 500 ktpa of feed from Maud Creek. The addition of a flotation plant enables campaign treatment of sulphides and is flexible in respect to treatment of oxide and transitional material. The Union Reefs plant is currently operated on a 9 days on 5 days off roster, with the crushing circuit utilisation lower again and typically only with one of the two mills operating (the second mill is only required approximately 10% of the time).

There are a number of advantages associated with leveraging the existing Union Reefs Processing Plant rather than building a standalone Processing Plant at Maud Creek including;

- Lower processing and infrastructure capital costs;
- Lower first fill costs;
- Low additional sustaining capital costs;
- Simpler and quicker project implementation, reduced technical risk;
- Lower water demand at Maud Creek;
- Less onerous approvals for existing processing facility;
- Lower overall operating cost;
- Production creep opportunities with existing facility;
- Simple and cheap future expansion to meet any increase in mining production;
- Ability to process oxide mineralization, so increasing the LoM of Maud Creek;
- More flexibility in processing, including transitional mineralization being able to be processed through the cyanide leach or flotation circuits or possible both, (higher gold recovery);
- Extensive tailings storage capacity already available (lower capital cost); and
- Extend the LoM of the Union Reefs/Cosmo underground mine operation.
- The key additional cost is haulage of mineralization. Other key considerations include the required approvals required for haulage and obtaining the social licence to operate.

17.2 Processing Plant Basis of Design

This study is based on historical testwork data and previous engineering studies provided by Newmarket Gold in consideration of the existing Union Reefs Processing Plant. The Project assumption is to process mineralization through a modified Union Reefs Processing Plant. A detailed integration study has not been undertaken into the Union Reefs processing option. This will be undertaken at the next Phase of the Study. However it is expected to be relatively simple, quick and straight forward. It presents an attractive opportunity.

Preliminary process design criteria (PDC) and process flow diagrams (PFDs) have been prepared as appropriate for this stage of study. Certain assumptions have been made based on incomplete knowledge of the metallurgy and market conditions. These assumptions are discussed in the 'Process Flowsheet Selection' section below. The typical engineering deliverables will be refined as part of future study work and will better reflect integration of Maud Creek mineralization into operations at Union Reefs.

A processing mass balance model for a standalone plant has been constructed showing flow rate/ tonnage and composition of all key streams. Equipment sizing and power draw calculations are performed within the model using the parameters specified in the PDC. The model inputs (PDC) and outputs are provided in the appendices.

The capital cost estimate has been built up from supplier estimates and the Simulus database for major mechanical equipment items, with estimates for minor equipment and fabricated items taken from the Simulus database. Mechanical equipment that already exists at the Union Reefs processing facility was then removed from the estimate. Earthworks, civil, structural, instrumentation and electrical disciplines have been factored from mechanical equipment costs. These factors have been adjusted to account for the existing site, systems, cleared areas, etc.

Other factored direct capital cost items include first fills, critical spares and warehouse inventory, laboratory equipment and supplies. Indirect costs are also factored. They include site temporary facilities, mobilisation and demobilisation, freight, vendor representation and site commissioning and EPCM. Other factored items already existing at Union Reefs such as site buildings, IT, communications and network equipment have been completely removed for these costs. A contingency of 10% has been included in the overall capital cost estimate. The capital cost estimate will be refined as part of ant future work with a focus on equipment sizing and costing and integration into the existing processing facility. It is likely there is still some double up of costs and that it can be reduced further because of the preliminary level of the costing.

Operating costs have been developed from first principles. Reagent consumption and grinding media rates are taken from the mass balance outputs and testwork. Unit costs are based on supplier quotes provided to Simulus Engineers by vendors during the previous six months. Labour requirements have been estimated based on the existing Union Reefs organisational structure assuming an extra shift of operators and maintenance is required as well as an extra flotation team member per shift. It uses recent industry assessment for salaries but will be updated at the next phase of study to use the actual salary structure at Union Reefs. Power consumption has been built up from the equipment list and power price based on the local grid prices. The operating cost estimate will be refined as part of future work. The bulk of the fixed costs have been removed as they are largely accounted for already through the Union Reefs operating costs. A future review may choose to share the fixed cost savings benefits between the Cosmo underground ores and Maud Creek mineralization.

The target accuracies for this stage of study work is +/-30% for capital cost and +/-20% for operating cost, based on the process design criteria as developed. There is potential for costs to rise or fall in future in line with future moves in exchange rates, equipment, material and labour prices.

17.2.1 Process Flowsheet Selection

Several earlier studies were used in conjunction with the testwork reports in as part of the process flowsheet selection. In particular:

- 1998 Signet Engineers “Process design Criteria 3081- G-00-F-001 Rev A” (Draft)
- 1997 “Review of the Metallurgy, Capital Cost and Operating Cost for the Maud Creek Project”, Signet Engineering May 1997
- 1998 “Metallurgical Evaluation of the Maud Creek Project” by John MacIntyre and Associates Pty Ltd. Includes abridged process design criteria

Based on these reports as well as a review of the testwork demonstrating the poor response to conventional cyanide leaching, initially a simple single stage crushing and closed circuit SAG milling circuit, followed by centrifugal style gravity concentration, simple rougher / scavenger / scavenger cleaning flotation circuit followed by concentrate dewatering was chosen as the base case flowsheet. Once the decision was made to utilise the spare capacity at Union Reefs Processing Plant, the front end crushing and grinding circuit design reverted to the existing 3 stage crushing and closed milling circuit configuration existing at Union Reefs. The process overview diagram is shown below in Figure 17-1. It highlights the new areas required as part of the flotation circuit upgrade modifications. The preliminary plant layout is provided further below in Figure 17-3. Downstream processing options were only considered at a preliminary level.

The Union Reefs Process Plant was originally designed to process free milling but predominantly fresh rock ores from the Crosscourse Pit with a BBMWi of 16 - 18 kWh/t at a grind size of 106 microns. Annual processing rates of 2.8 Mtpa (~140 t/h Mill 1 and 190 t/h Mill 2 at ~300% recirculating load) were achieved between 1999 and 2003. These ores typically had an average gold head grade of 1.5g/t recovering 92 - 94%, of which 30% was recovered by gravity. Union Reefs is significantly oversized for the current feed from the Cosmo Underground Mine. The spare milling capacity presents the key opportunity to Newmarket Gold.

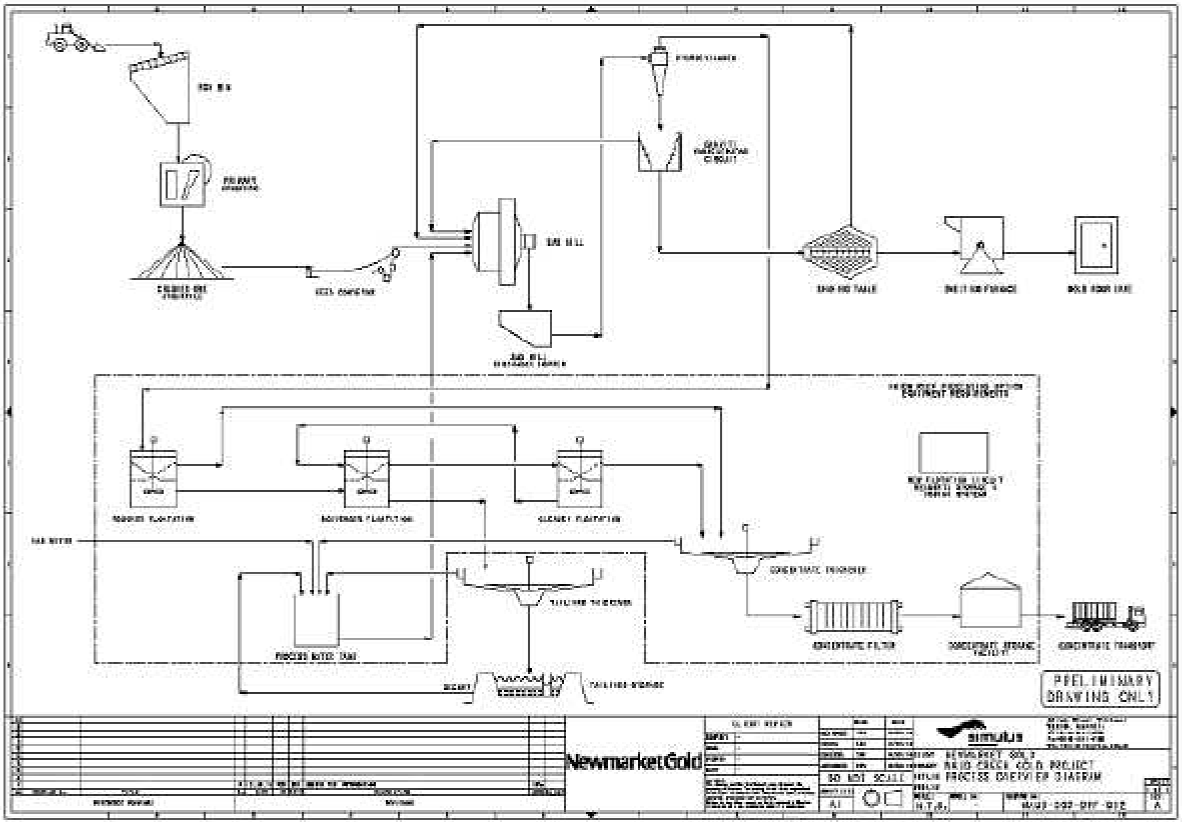


Figure 17-1: Maud Creek Gold Project Process Overview Diagram

At the Union Reefs Processing Plant, the ore is crushed and then milled in closed circuit with hydrocyclones. A separate gravity circuit recovers gravity recoverable gold from the ball mill discharge. The fine milled product from the cyclone overflow is pumped to a new flotation circuit where a high grade low tonnage gold bearing concentrate is produced from the sulphide minerals leaving a gangue non-sulphide residue that can be disposed of, to tailings. The concentrate is dewatered through thickening and filtration before being bagged, stored in shipping containers and transported by road, then ship to customers in China.

The selected grind size is 80% passing 75 microns. This was chosen following the 1996 optimisation testing, although earlier pilot programs used a coarser size, (125 microns in 1996 and 100 microns in the 1997 program). Further optimisation testing is recommended to confirm the grind size however the Union Reefs milling circuit will be flexible to a range of grind sizes.

The evaluation conducted by J MacIntyre in 1998 included a detailed economic optimisation of the grind size, as shown below in Figure 17-2.

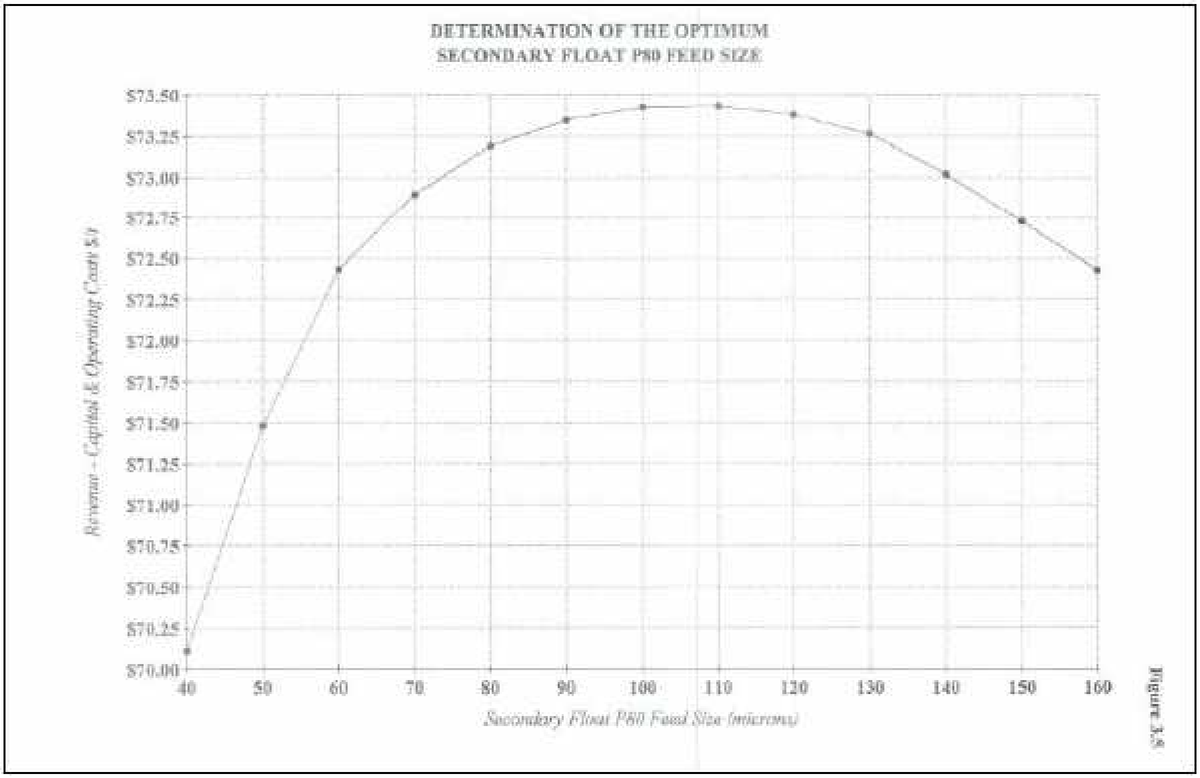


Figure 17-2: Economic Optimum Grind Size

Source: J MacIntyre, 1998

Although 100 - 110 microns was determined to be the optimum grind size, a P80 of 75 micron was judged more appropriate for design purposes. The slightly higher capital cost would be offset by better operating flexibility and lower risk.

Several important design parameters have changed since this evaluation was conducted:

- Circuit tonnage - was 300 ktpa now 500 ktpa
- Ore grade - was 7.4 g/t now approximately 4.38 g/t
- Comminution circuit remains three stage crushing + ball milling
- Downstream processing - was Bio-oxidation now direct concentrate sale
- Gold price - was AUD482/oz now >AUD1,500
- Power cost - was AUD0.125/kWh, now AUD0.22/kWh
- At that time concentrate was expected to be processed locally. With the base case of processing the concentrate at a remote location and the subsequent impact of concentrate transportation costs, a finer grind is further supported (higher concentrate grade and reduced mass pull).

The gold price appears to be a significant change. A higher price should mean a finer economic grind size, all other things being equal. Therefore there is unlikely to be a strong case for a coarser grind. On the other hand, recovery does not increase much at finer sizes. Grinding from 80 - 40 microns would increase gold recovery by about 0.8%, according to the projection model used by MacIntyre.

At this stage it is recommended to leave the design grind size at 75 microns. Further economic optimisation is recommended in the later stages of design. Processing through the Union Reefs milling circuit will be flexible to a range of grind sizes and this argument is largely redundant for design purposes. It can be optimised during operations.

The grinding circuit includes gravity gold recovery on a separate mill discharge pump. Gravity testing gave mixed results and the impact on overall recovery is not well established. However it does insure against coarse gold losses and enables on-site bullion production via smelting of the gravity gold concentrate.

The selected flotation circuit includes rougher, scavenger and scavenger cleaner stages, but no flash flotation. Some of the testing showed marginal benefit from flash flotation. It was found that the existing roughers in combination with the gravity circuit were highly effective, and addition of flash flotation is considered an unnecessary complication. However a flash flotation cell(s) could be incorporated if required. It will be considered further at the next stage of study.

A cleaner flotation stage will increase concentrate grade but possibly marginally reduce recovery. The economic trade-off will be affected by the choice of downstream processing route chosen, or direct concentrate sale. The cleaner circuit is included in the design as per the pilot plants, so product grade projections are consistent with the test results. It may well be possible to achieve the target 45g/t gold in concentrate without a cleaner circuit, but this circuit is fairly cheap in both operating and capital cost. Regrind of the scavenger concentrate might also be considered but has not been tested at this stage.

The flotation reagent scheme consists of sodium isobutyl xanthate (SIBX, collector), copper sulphate ($CuSO_4$, activator) and frother, with no pH adjustment. This scheme is commonly applied in flotation of gold bearing sulphide mineralization.

Given the high combined flotation and gravity recovery of gold, leaching of the flotation tails is unlikely to be economically viable. Earlier work recommended a tails leach circuit, but this was in the context of an integrated process plant including oxidation and cyanide leach of concentrate. With processing through Union Reefs, this operational flexibility is now an option to improve recoveries on transitional mineralization. Flotation tails can therefore report either to the CIL circuit or directly to tailings storage (via a tailings thickener due to water balance and environmental discharge considerations).

The gold flotation concentrate is thickened and filtered for transport. The gold is refractory and requires some form of oxidative treatment. The carbon content also creates preg-robbing characteristics. Circuit selection for the downstream process depends on a number of issues such as:

- Concentrate mineralogy, especially sulphur, arsenic, carbon and gold content
- Local economic factors, e.g. cost of power, reagents
- Environmental regulations
- Technology risk profile.

Downstream processing is outside the scope of this study. Some discussion of direct concentrate sale and other alternative downstream processing options available is provided in Section 17.7.

17.2.2 Plant Throughput Selection

Both the mining inventory and life of mine (LoM) need to be considered to give a reasonable plant throughput, with the plant sized initially largely driven by the desired life of mine. This in turn must be balanced against the operating and capital cost economies of scale and any financing obligations.

The preliminary underground mining inventory used for the plant sizing is approximately 2.5 Mt at 4.4 g/t Au, and the open pit mining inventory is 230 kt at 5.6 g/t Au. Both of these values are based on a cut-off grade of 1.5 g/t. While there is potential to grow the mining inventory, the plant sizing was undertaken based on the tonnes currently available at the time. This sizing now applies to just the flotation plant. SRK notes that significant increases in flotation capacity can be achieved with very low incremental increases in capital cost. If underground mining production is able to be increased, this provides a good opportunity to improve the project returns.

Based on a five year LoM, a 500 ktpa plant is recommended as the base case. A 500 ktpa plant size strikes a good balance between the current mining inventory, potential LoM extension, capital and operating cost increases, while maintaining a lower cost modular plant supply and construction philosophy.

The 500 ktpa throughput rate is further supported by the likely underground mining rates. Since the deposit is relatively narrow, a 500 ktpa production rate is a reasonable assumption. A higher mining rate would be challenging given the likely method and geometry of the deposit. The process plant can be simply debottlenecked (for example equipment at this throughput is often oversized) to achieve additional throughput if mining rates and LoM allow it in the future; this will be assessed in more detail in the full feasibility study. This LoM also allows the owner to economically amortise the

The key process design criteria used for this PEA is shown in Table 17-1. Justification and explanations are provided through the preceding and following sections.

Table 17-1: Key Process Design Criteria Information

Variable	Unit	Value
LoM tonnes	Mt	3.8
Mineralization type	#	96% Fresh, 4% Oxide/ Transitional
Mill Tonnes	tpa	500,000
Plant utilisation	%	90
Milling throughput	tpa	70
Gold Feed Grade	g/t	4.38
Gravity Recovery (fresh)	%	20
Flotation Recovery (fresh)	%	75
Total Recovery (fresh)	%	95
Total Recovery (oxide/ transitional)	%	85
Recovered Gold	oz/a	66,890
Concentrate Mass Recovery	%	7.3
Concentrate Production	tpa	36,500
Concentrate Gold grade	g/t	45
Concentrate moisture	%	10
Tailings	tpa	463,514
Water Consumption (plant)	GL/a	0.35

17.3 Engineering Deliverables

A number of the key process engineering deliverables have been completed at a preliminary level for the Project. The deliverables were derived in two phases; the first phase involved engineering associated with a standalone plant located at Maud Creek and the second phase made allowance for integration and modifications to the existing Processing Plant at Union Reefs that is now the Project base case assumption.

All engineering deliverables would be updated during future phases of work.

17.3.1 Processing Description

The Processing Plant description is provided below along with some indicative drawings of the plant and the proposed modifications. The main stages of the process consist of:

- Three stage crushing and product screening circuit (existing)
- Two closed circuit ball mills, each with a hydrocyclone (existing)
- Gravity circuit including Knelson concentrators (existing)
- Flotation circuit (new)
- Concentrate thickening and filtration circuit (new)
- Concentrate storage and loading (new)
- Tailings thickener (existing)
- Utilities (largely existing)
- Reagents (largely existing).

It is expected that there will be some additional overlap as well as some additional modifications required to the existing plant. Confirmation is required to whether a separate process water circuit is required due contamination of the CIL circuit with flotation chemicals and vice versa. For example cyanide is often used as a depressant in pyrite flotation and this could impact gold recovery. It has been allowed for in the capital cost estimate. A plan drawing of the overall Processing Plant and Union Reefs with the proposed location of the flotation circuit is shown below in Figure 17-3.

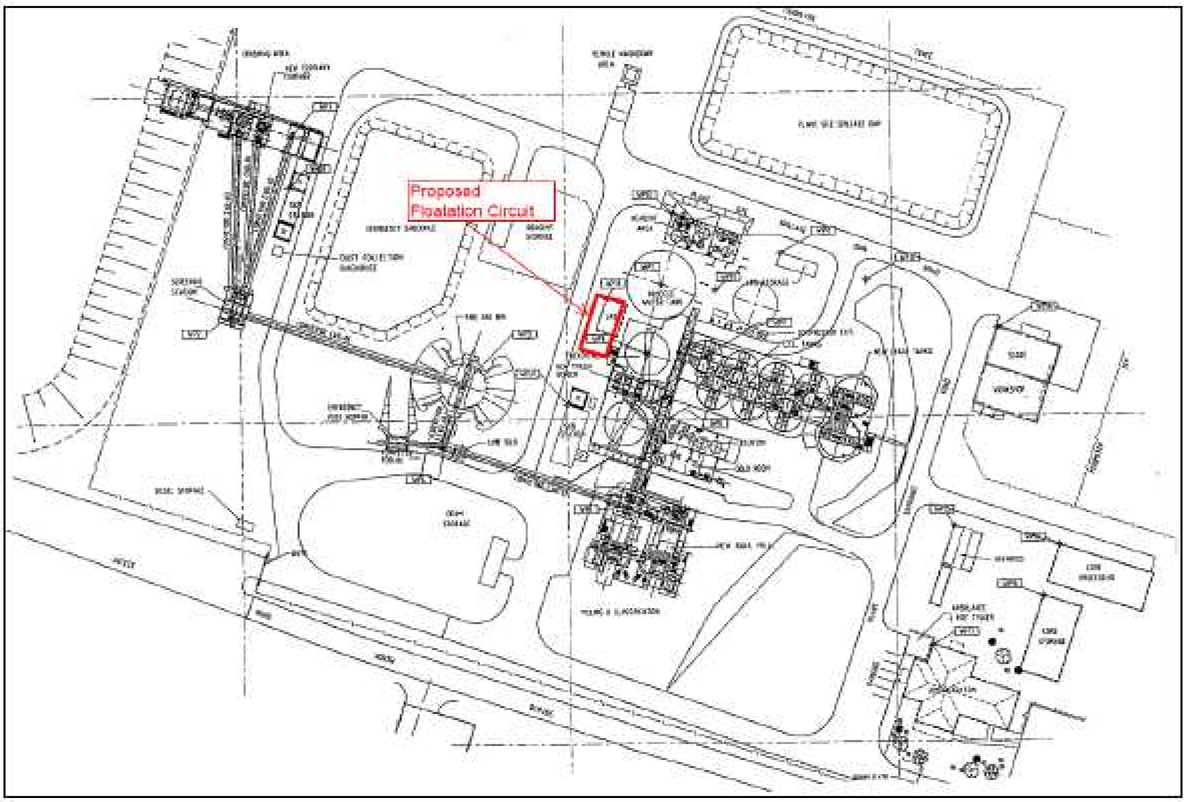


Figure 17-3: Union Reefs Site Plan

17.3.2 Crushing and Screening Circuit

The Unions Reefs crushing circuit is shown diagrammatically in Figure 17-4.

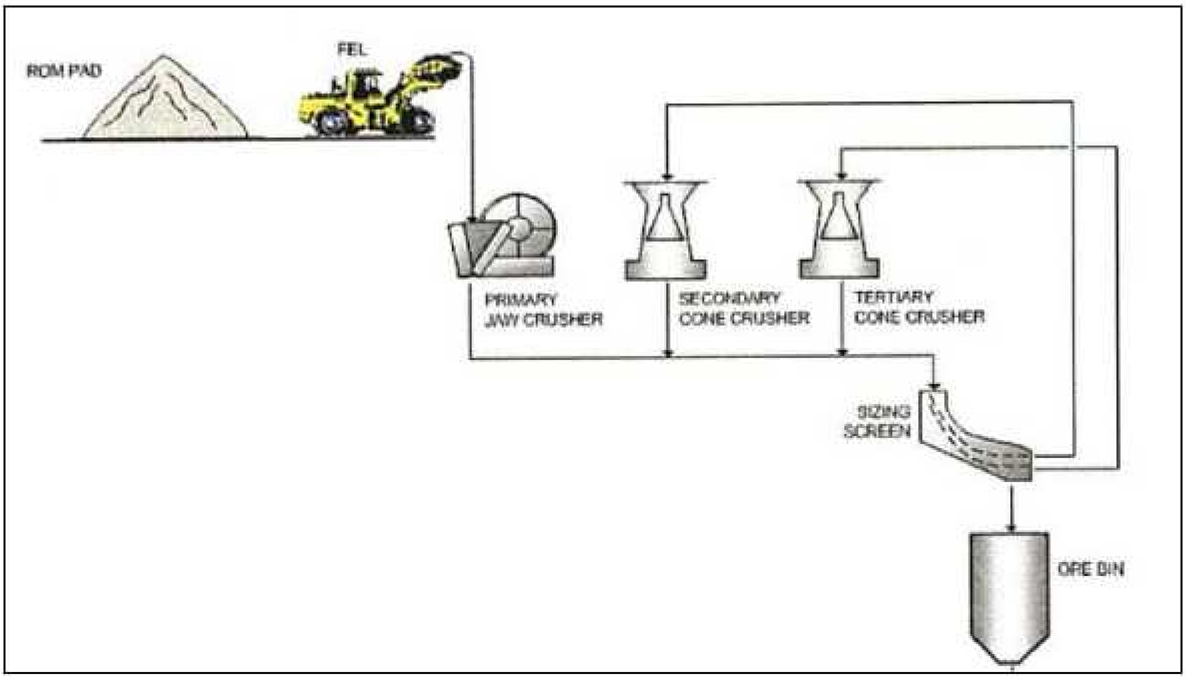


Figure 17-4: Crushing Circuit at Union Reefs Processing Plant

Run of Mine material is reclaimed from stockpiles by a wheel loader and tipped into a ROM bin. A vibrating feeder transports material from the ROM bin to a C140 Nordberg Single Toggle Jaw Crusher. Any oversize rock is broken at the Jaw Crusher by a hydraulic rock breaker. Jaw Crusher product is transported to a double deck "banana" 3.1 m x 7 m Nordberg Product Screen with a 40 mm aperture top deck and a 14 mm aperture bottom deck by conveyor belts. Tramp steel is removed after the jaw crusher by a fixed magnet to protect the secondary cone crusher and conveyor belts. Screen undersize (minus 14 mm) product is conveyed to a Fine Ore Bin/ Stockpile of 3000 tonnes live capacity, while screen oversize (plus 40 mm product) is transported to a Nordberg Omni-Cone 1560 secondary cone crusher. Secondary crusher product returns to the Product Screen. Intermediate product of minus 40 mm plus 14 mm from the screen is transported to a Nordberg HP500SX tertiary cone crusher. Tertiary crusher product is also returned to the product screen. No changes are envisaged to the existing crushing circuit.

17.3.3 Milling Circuit

Fine crushed ore is reclaimed from the fine ore bin by a slot type belt feeder. Stockpiled fine ore from outside the bin can be reclaimed using a wheel loader which tips to a day (emergency) feeder bin. Both the slot feeder and day feeder discharge to a single mill feed conveyor. The mill feed conveyor discharges to a Mill Feed hopper. The Mill Feed hopper has a split discharge onto two variable speed ball mill feeder conveyors, one feeding the 3 MW, 4.7 metre by 8.2 metre ANI No.1 Ball Mill, the other feeding the 4 MW 5 m x 9.1 m ANI No.2 Ball Mill. Each mill is able to be operated independently and/or in isolation, and each is in closed circuit with separate hydrocyclone clusters.

Mill discharge slurry overflows through a trommel screen into a discharge hopper, from where a portion is pumped to the hydrocyclones, a portion to the centrifugal gravity concentrator and the oversize scats material reports back to the ball mill.

Cyclone underflow is returned to the ball mill feed chute while the cyclone overflow reports to the flotation feed tank. The milling circuit at Unions Reefs is shown diagrammatically in Figure 17-5.

No changes to the existing milling circuit are envisaged.

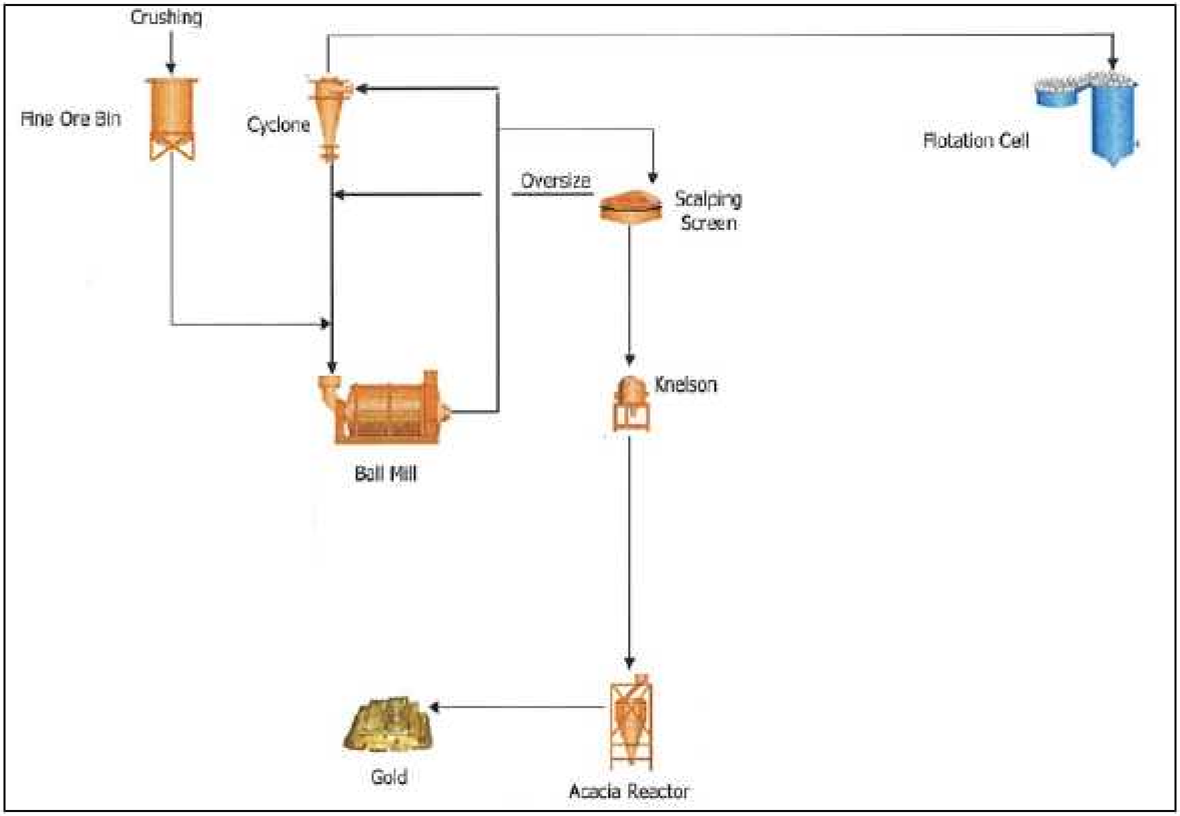


Figure 17-5: Grinding and Gravity Circuits at Union Reefs Processing Plant

17.3.4 Gravity Recovery Circuit

Each Ball Mill hydrocyclone cluster is fitted with a direct off-take on the hydrocyclone feed pot, which directs a bleed stream of ball mill discharge to a Nordberg 1.2 metre by 1.5 metre scalping screen to remove coarse scats. Fine screen product reports to one of two automatic discharge 30 inch Knelson concentrators. Rough Knelson concentrate containing coarse gold is automatically discharged to a secured hopper located within the gold room.

On a batch basis, the rough Knelson concentrates are transferred from the secured hopper to an Acacia intensive leach reactor (ILR). The concentrates are deslimed using water before caustic cyanide leach solution containing a proprietary “Leach Aid” is recirculated through the concentrate bed to rapidly dissolve the contained coarse gold. At the completion of the leach cycle, the concentrate residue solids are washed and discharged back to the grinding circuit. The high grade pregnant leach liquor containing the gold is then recirculated through an electrowinning cell, with the gold won onto stainless steel mesh cathodes. The high-quality won gold is then direct smelted to saleable bullion product.

The gravity recovery circuit at Union Reefs is shown diagrammatically in Figure 17-5. No changes to the existing gravity circuit are envisaged.

17.3.5 Flotation Circuit

Cyclone overflow is first screened to remove trash, then flows to the conditioning tank, where copper sulphate, frother and SIBX (xanthate collector) are added and given time to mix. Copper sulphate and SIBX can also be added upstream to the mill circuit if this is found to be beneficial.

Conditioned concentrate overflows to the rougher flotation cells. Rougher concentrate reports directly to the final concentrate thickener. Rougher tails flows to the scavenger cells.

Scavenger cells flotation concentrate is pumped back to the cleaner flotation cells, while scavenger tails is pumped to the tailings thickener.

Cleaner concentrate reports to the final concentrate thickener. Cleaner tailings are usually returned to the rougher cells, but may also be returned to the scavenger cells. The next stage of design will also enable tailings to be pumped to the CIL circuit to allow for low recovery Maud Creek transitional mineralization. Refer to Figure 17-6 for an isometric overview of the flotation area.

17.3.6 Concentrate Thickening Filtration Circuit

Flotation concentrate is first settled in a high rate thickener. The thickener underflow is further dewatered in a filter press as per conventional practice. It is dumped to ground and contained in a storage shed where it can be bagged, then loaded into sea containers and onto trucks. Filtrate and thickener overflow are returned to the process water tank. A separate process water system has been allowed for at this stage of study to eliminate concerns with reagent contamination between the CIL and the flotation circuits. Figure 17-7 provides an isometric overview of the concentrate filtration, concentrate storage and bagging area.

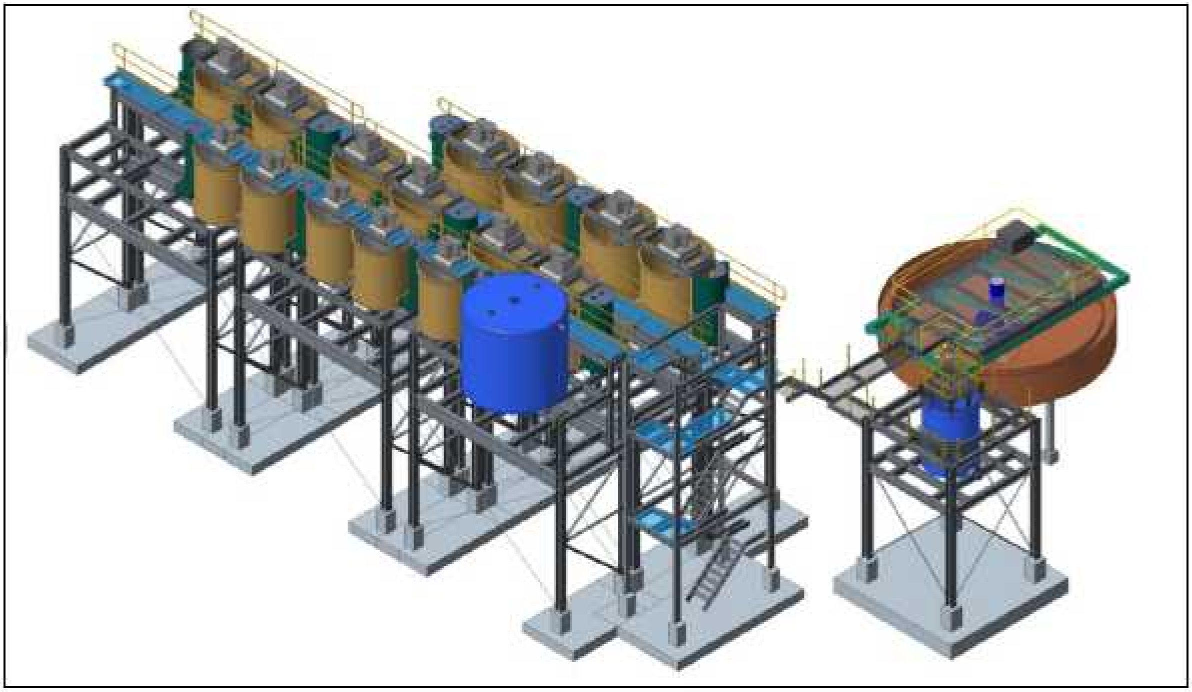


Figure 17-6: Proposed Flotation and Concentrate Thickening Isometric Drawing

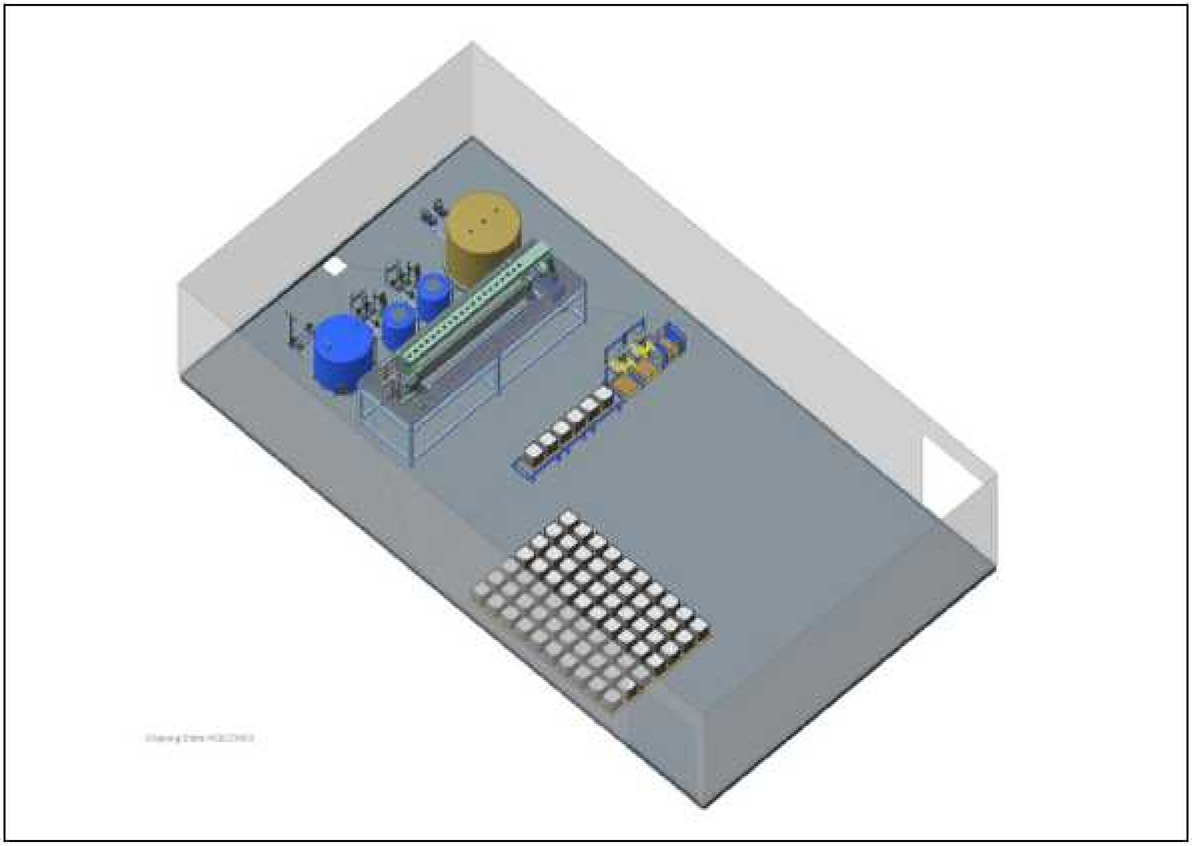


Figure 17-7: Concentrate Filter, Storage and Bagging Area Isometric Drawing

17.3.7 Tailings Circuit

Flotation tailings are thickened and the underflow pumped to the tailings storage facility. The overflow is returned to the process water tank. The next stage of design will also enable tailings to be pumped to the CIL circuit to allow for low recovery Maud Creek transitional mineralization.

17.4 Reagents and Services

17.4.1 Reagents

Xanthate, copper sulphate and Flocculant are delivered to site in solid bulk bags. They are mixed with water in batches, then pumped to where they are required.

Frother is delivered to site as a liquid, stored in a tank and pumped to the flotation area.

Grinding media is provided in 44 gallon drums or in bulk sea containers where it is unloaded into concrete bunkers.

Major reagent consumption expectation based on testwork is shown in Table 17-2.

Table 17-2: Reagent Consumption Rates

Reagent	Consumption Rate (g/t)	Consumption Rate (tpa)
Frother	20	10
Collector (SIBX)	125	82.5
Activator ($CuSO_4$)	50	16
Flocculant	70	16.6
SAG Mill Grinding Media	2810	1405

17.4.2 Water

The existing facilities at Union Reefs Processing Plant will be used to draw the required raw and potable water.

Process water consists of thickener overflows, tailings dam reclaim water, and brine from the water treatment plant, with make-up from the raw water. A separate process water system has been allowed for at this stage of study to eliminate concerns with reagent contamination between the CIL and the flotation circuits. Confirmation is required to whether it is required and the level of associated risk. It has been allowed for in the capital cost estimate.

Additional discussion on water is provided in Section 18.6.

17.4.3 Air

Plant air will be supplied by a compressor and air receiver package. Instrument air will be supplied by plant air ran through a drying and air receiver package. Low Pressure (LP) flotation air will be supplied by three air blowers running 2 duty / 1 standby. The full plant and LP air requirements has been allowed for in the capital costs although it is expected there will be some overlap and capital costs can be reduced at the next phase of study.

17.4.4 Power

Power is provided by an incoming line from the Northern Territory grid. Additional information on power supply and reticulation is provided in Section 18.3.

17.5 Flotation Concentrate

17.5.1 Flotation Concentrate Grade

Flotation concentrate composition from the pilot plant runs was reported as shown below in Table 17-3:

Table 17-3: Pilot Plant Flotation Concentrate Composition

Tests	Au g/t	S%	As%	Fe%
1996	47.5	17.2	3.1	20.8
1997	51.4	19.5	3.8	21.9
2006	40.7	22.2	3.9	-
Average	**46.5**	**19.6**	**3.6**	**21.3**

Comprehensive analysis of the concentrates was not undertaken for all pilot campaigns. Additional assays from the 1996 campaign are presented below (indicative only).

Ag - 20 g/t

Cu - 1600 ppm

C(total) - 2.79%

C(organic) - 0.04%

SG - 3.2.

An XRD analysis on a combined concentrate from the same piloting campaign is shown in Table 17-4 providing approximate mineral proportions based on visual estimates.

Table 17-4: Pilot Plant Flotation Concentrate XRD Analysis

Mineral	**%**
Pyrite	40
Arsenopyrite	25
Chalcopyrite	3
Marcasite	3
Galena	<1
Sphalerite	<1
Goethite	1
Quartz	10
Muscovite /Sericite	10
Chlorite	2
Carbonate (dolomite)	6
Total	**100**

This is the best available indicator of what concentrate specification is likely to be produced from the flotation plant. It does not include gold recovered from the gravity section. A gravity concentrate grade of 45 g/t Au has been selected for design purposes and transportation costs of the concentrate. This is considered to be conservative. It is expected cleaning of the concentrate, flash flotation and/or a finer grind will improve the concentrate grade at similar recovery. Metallurgical testwork will be required to generate a flotation concentrate sample to provide to customers for their own testing.

Note that the first two pilot plants included periods with and without cleaner flotation, while the design includes a cleaner stage.

17.5.2 Concentrate Sale Options

The likely market for the gold concentrate is China. This has been selected as the base case. Within Australia, there are a few operations that could potentially take the concentrate, but transport costs are high and at first inspection, it has been assumed that the customers do not have sufficient capacity available to treat the full production tonnage. These options are discussed further in Section 17.7.

Newmarket Gold' s Fosterville Gold Mine in Victoria could potentially treat the concentrate through its BIOX® circuit. Biomin (the BIOX® technology providers) has indicated that this is technically feasible and extensive testwork and engineering study has been undertaken on this option. However there is insufficient capacity for all the concentrate, the transport cost is likely to be quite high, and recovery would be affected by the preg-robbing characteristics. The scope of the study specifically excludes this as a sale option at this time.

17.5.3 Concentrate Terms

The payable gold content depends on the tonnage sold as well as gold and impurity grades.

Under Chinese regulations, the gold concentration must be over 40 g/t to be classified as a concentrate. Most metal sulphide concentrates imported into China have arsenic importation limits of 0.5%, gold concentrates do not have this limitation. The arsenic grades in the Maud Creek gold concentrate are well above this level (approximately 3.6% based on piloting) as shown in Table 17-3. Lower gold grades are classed as ore and incur higher taxes and arsenic grade is likely to result in importation restrictions. Maud Creek concentrate should have no difficulty exceeding the 40 g/t threshold.

The major impurity to consider is arsenic. Chinese customers will accept material in the 3 - 5% range expected from Maud Creek, but it incurs a penalty based on the grade.

A number of indicative terms have been provided for the Maud Creek concentrate, the best of which would realise about 95% payable gold content, with the following deductions:

Treatment charge (TC) AUD10 / tonne concentrate

Up to 97% could be realised with higher gold grades of >60 g/t. The arsenic penalty is expected to be AUD4/t concentrate for every 0.1% arsenic over 0.2% arsenic in concentrate.

This is based on informal discussions with market players and experience on similar projects, and closely agrees with values used in the Core Process Engineering report 140-001 of September 2011. Further benchmarking has been undertaken. The next phase of study needs to progress concentrate sale and terms negotiations.

17.5.4 Transportation costs

This is discussed in Section 19.1 Concentrate Transport.

17.6 Process Processing Risks and Opportunities

A number of process risks and opportunities have been covered in detail throughput the report. The risks are largely mitigated due to the significant testwork that has been undertaken on the Maud Creek mineralization, the favourable metallurgical behaviour and the exiting Union Reefs Processing Plant available to process the Maud Creek mineralization. A simple conventional flowsheet is required. A summary of the key process risks and opportunities are given below.

17.6.1 Risks

Metallurgical samples were biased to shallower samples.

The mineralization is hard and abrasive, which may present a number of issues. These include high media consumption, increased milling power requirements, and mill liner wear, which will drive up operating costs.

The gravity testwork gold recovery is highly variable, however overall gold recovery remains high.

There is a lack of information on flotation testwork performed on the transitional mineralization. Although the transitional mineralization only accounts for around 11 percent of the total mill feed, it will still have an effect on the project value. It is recommended some further flotation testwork be done on the transitional mineralization.

Development of feed and concentrate grade versus recovery relationships needs to be developed. This may require testing on some lower grade samples (testing has been biased to higher grades).

With the change to Union Reefs as the base case processing option, additional review of the oxide mineralization behaviours is required although it makes up a relatively small proportion of the overall feed.

There are a number of downstream processing options available. More work is required at the next phase of study to confirm that the direct sale of concentrate remains the most favourable. The risk of future curtailments of gold concentrate into China (possibly through tightening of the arsenic importation limits) warrants further consideration.

17.6.2 Opportunities

There has been limited metallurgical testwork done on flash flotation. The piloting that has been done showed it was effective but it has not yet been included in the flowsheet. There may be potential increase recovery and/or concentrate grade with a marginal capital cost increase. This remains a flowsheet opportunity and will be examined in more detail during future phases of work.

Finer grinding may improve plant performance. As with the other opportunities, this should be examined in more detail in the next phase of the study.

There may be an opportunity to increase concentrate grade but at a marginal reduction in gold recovery. Depending on the final downstream processing option selection, the recoveries associated with a higher concentrate grade may be justified by the reduction in concentrate transportation costs.

The use of the Union Reefs Processing Plant allows for potential increases in the throughput.

17.7 Toll Treatment Options for Product Concentrate

The Maud Creek deposit contains refractory gold mineralization, as demonstrated by testwork. This refractory nature is due to the close association between the gold and sulphide mineralization. Complicating things is the presence of carbonaceous mineralization presenting 'preg-robbing' behaviour during cyanide leaching, resulting in the mineralization being classified as 'double-refractory'.

Due to the gold being sulphide locked, it requires alternative treatment to direct conventional cyanide leaching. The relatively small production rate for Maud Creek concentrate and short LoM makes the construction of a dedicated refractory gold treatment plant for extraction of gold uneconomic. The best option is to produce a gold concentrate for treatment at a separate facility, which opens up the options to all cyanide pre-treatment technologies and other processing methods for recovering gold from refractory mineralization. Starting with the "Maud Creek Gold Project Processing Options Review" undertaken by Signet in 1998, each of the major pre-treatment technologies has been considered in the past. This latest assessment revisited these previous studies and undertook a preliminary review of the best options available to the Project, including potential 3rd party processing options in Australasia and abroad. Brief descriptions of the main options are provided below including the initial findings.

17.7.1 Ultrafine Grinding

Ultrafine grinding (UFG) involves the use of a stirred medium grinding mill to grind and liberate locked gold from sulphides by grinding as fine as 1 - 10 µm. The oxidised mineralization can then be treated by conventional cyanide leaching. There is generally an economic optimum point between cost and recovery resulting in a grind coarser than this. Ultrafine grinding was tested previously and found to be relatively ineffective. In the Signet study, median gold extraction rates of 61% at 100% -125µm only increased to 65% at 100% -20 µm. In a later report (by Hydromet Innovations for Mercator Gold, 2007), UFG to P80 of 15 µm increased extractions by between 6 and 22% of total gold, to values in the range 66 to 81%. In the 81% extraction case, gold extraction was increased to 86% by adding alkaline pre-treatment, but this would require a toll treatment operator to tailor their process to incorporate this pre-treatment. For this concentrate alkaline pre-treatment would likely be infeasible for the available benefit. Gold extraction of 71% was reported from UFG in a report by Independent Engineers (Interim Findings Fatal Flaw Review Maud Creek Project, 2005). Neither does it account for the preg-robbing of the mineralization. Collectively, these results make UFG unattractive based on the cost versus increased gold extraction and recovery.

17.7.2 Bio-oxidation

Bio-oxidation uses microbes to oxidise the gold containing sulphide matrix, breaking down the sulphide matrix surrounding the gold particles and exposing them for further treatment. The oxidised mineralization can then be treated by conventional cyanide leaching. The Signet tests were successful in achieving extraction rates up to 95%, although there were some challenges with regard to high carbonate and arsenic (As) levels, which would have to be resolved for effective implementation. The Hydromet Innovations study returned a gold extraction rate of only 72%, and 86% was reported by 'Independent Engineers' . The key barrier to pursuit of this option, however, is that of the three BIOX® plants identified in Australia, two (Beaconsfield and Wiluna) are in care and maintenance and are remote to the site; Tasmania and the northern WA Goldfields respectively. The one remaining operating site (Fosterville) has been assessed in the past. It does not have sufficient capacity for the entire production rate. Fosterville is owned by Newmarket Gold and they could bring considerable technical expertise to this option however a standalone BIOX® plant is not justified. Part off-take could be future opportunity but will need to consider the amount of preg-robbing material in the feed.

17.7.3 Pressure Oxidation

Pressure oxidation is a process where sulphur removal is carried out under high pressure and elevated temperature in an autoclave. The oxidised mineralization can then be treated by conventional cyanide leaching. The Hydromet Innovations study reported POX extraction of 98%. However, there are no POX autoclaves available for gold concentrate in Australia. Macraes (New Zealand) is an option and it is understood it is prepared to accept some concentrate, but the indicative treatment terms provided for other operations were marginally less favourable than those for the sale of concentrate to Chinese processors and has not been pursued further at this stage. It remains and option. It is a good option for arsenic containing concentrates as it produces a relatively stable precipitate.

17.7.4 Ultrafine Grinding followed by Moderate Pressure Leaching (Albion/Activox/CESL)

The Albion, Activox and CESL processes all involve oxidative leaching at atmospheric (Albion) or under moderate temperature and pressure conditions (Activox and CESL), with oxygen injection (Albion and Activox) or oxygen and HCl addition (CESL) at elevated temperatures (100^{o}C for Albion and Activox, 150^{o}C for CESL). The oxidised mineralization can then be treated by conventional cyanide leaching. A report by Independent Engineers (Interim Findings Fatal Flaw Review Maud Creek Project, 2005) indicated gold recoveries of 86% for Activox alone, and for the combined Albion / CESL processes. An Activox test in the Hydromet Innovations report achieved gold extraction of 92%. These processes are capital cost inhibitive due to the short life of mine and low tonnages.

17.7.5 GEOCOAT®

The Geocoat® process involves coating a carrier rock with concentrate slurry, then stacking the material onto an impervious pad and performing bio-oxidation. Independent Engineers reported gold extraction of 81%, and poor performance on a number of other grounds resulted in this being rated as the lowest preference in the range of ten options in their assessment.

17.7.6 Roasting

Roasting oxidises the sulphide concentrate releasing SO2 gas. The oxidised mineralization can then be treated by conventional cyanide leaching. The KCGM owned Gidji roaster is shutting down and is not designed for arsenic in the concentrate, and is therefore not an option. The Kanowna Belle roaster can process arsenic contaminated concentrates, does currently have some available capacity but no formal approach has been made to ascertain how much. Transport costs (both in the vicinity of Kalgoorlie, WA, is an additional disincentive; therefore this option has been disregarded.

17.7.7 Selected Downstream Processing Option

Preliminary techno-economic assessment of the above options shows the terms offered by Chinese processors for direct smelting of the concentrate, which includes allowance for arsenic content, to be the most favourable option. This option is used by part of KCGM' s concentrate, Evolution Mining' s Mt Carlton Operation and KBL Mining' s Mineral Hill' s Pearse Gold concentrate. Further details of the preferred option are provided in Section 19. At the preliminary level of review, this option appears to offer the most favourable outcome, and is the primary one being pursued.

There may be an opportunity to use a mixture of the third party concentrate treatment options, due largely to available capacity, but this will make the downstream processing and sale process more complex. At this stage just the single sale option is being considered.

17.8 Stand-alone processing Plant Option

SRK completed a study to assess and define the requirements for the options of a Processing Plant on site at Maud Creek. The following is provided as background to the work undertaken to support the economic parameters presented in the comparison of the options. The Union Reefs Processing Plant option is the preferred case being presented in this PEA.

This option would require construction of a Tailings Storage Facility, increased water management, power and infrastructure requirements while providing paste fill for the proposed underground operations.

The current study reviewed the previous metallurgical testwork data and supporting engineering studies. From this body of work, a simple, conventional, inexpensive and low-risk flotation circuit has been selected, allowing flexibility with respect to circuit configuration and downstream processing. The base case presented is for a standalone mill and flotation concentrator plant at the Maud Creek site, producing a gold sulphide concentrate for sale or possible in-house processing. A gravity product will also be produced and either smelted on site or smelted and refined in Australia at one of the gold refiners.

The flowsheet consists of a single stage crushing and closed circuit SAG milling circuit, followed by centrifugal style gravity concentration, simple rougher / scavenger / scavenger cleaning flotation circuit followed by concentrate dewatering was chosen as the base case flowsheet. The inclusions of a pebble crusher and/ or a flash flotation would considered in a future work but have not been tested to the same extent as the flowsheet selected.

Downstream processing options for refractory gold mineralization and concentrates were considered at a preliminary level. The direct smelting of flotation concentrates option has been selected. This involves concentrate being dewatered, bagged, stored in shipping containers and transported by road, rail then ship to China. A plant capacity of 500 ktpa has been selected by SRK to support the mine production capability and as it strikes a balance allowing for a modular style of design and construction and management of the capital cost and reduce site construction.

17.8.1 Stand-alone Capital and Operating Cost Estimates

The stand-alone Processing Plant capital and operating cost estimates are summarised in Table 17-5 and Table 17-6.

Table 17-5: Stand-alone Processing Plant – Capital Cost Estimate

Item	Total (AUD M)
Process Plant	58.7
Surface Infrastructure	5.2
TSF Construction	12.0
Paste Fill Plant	4.0
Sustaining Plant	10.9

Table 17-6: Stand-alone Processing Plant – Operating Cost Estimate

	AUD/t
Processing	34.94
Concentrate transport	17.11
Site G&A, Indirects	13.00

18 Project Infrastructure

18.1 Site Access

A suitable access road connecting the Maud Creek project site to the Stuart Highway is required to facilitate access. The options are illustrated in Figure 18-1.

The existing unsealed access road to the Maud Creek site into Katherine, heading south from the mine was used during previous mine operations to haul material from site. The access road is linked to the Stuart Highway by Ross Road, a sealed public road which is regularly used by horticultural landholders in the area. The access road crossed Gold Creek just south of the existing mine pit and during the wet season, the road is often impassable and experiences significant damage.

The existing access track would need to be upgraded and have a crossing installed over Gold Creek to enable transport operations to continue during the wet season both into and from site. This route has been approved by the Northern Territory Government as a Right of Way and General Service easement. A second approved Right of Way and General Service easement runs north from the Mine to Gorge Road. This provides the site with a second access point to Katherine.

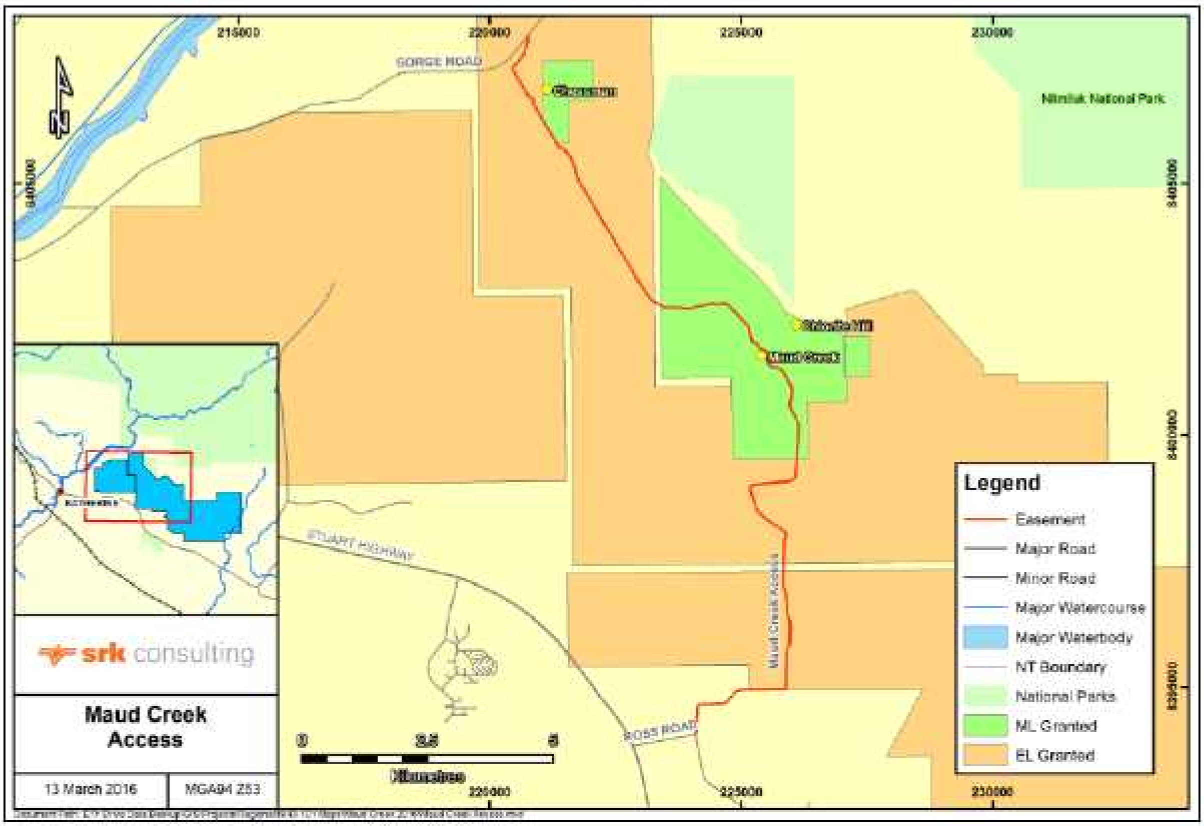


Figure 18-1: Site Access Roads

18.2 Surface Infrastructure

The Maud Creek Gold Project surface facilities are representative of a modern and conventional flotation style concentrators and underground mining operation. The site comprises the following:

- Office and administration complex, including change house;
- Store and laydown facilities;
- Heavy underground equipment workshop;

Temporary surface mineralization stockpiles and waste stockpile area;

Maud Creek Open Pit Mine and portal;

Ventilation exhaust raises;

Ventilation intake raise; and

Raw water storage to manage rainfall runoff.

18.3 Power Reticulation and Supply

Power demand estimates are based on the 0.5 Mtpa standalone processing case and are summarised in Table 18-1. Power requirements for the Processing Plant was estimated based on the process plant motor list (preliminary) and assumed power draw and utilisation. This included an allowance for administration buildings, workshops and lighting. This is now largely redundant as processing will be at the existing Union Reefs Processing Plant that already incorporates this. The overall Maud Creek demand is significantly reduced as a result. It impacts the selection of the preferred power source.

At the Maud Creek mine site, an allowance of 2 MW of power has been made for the underground mining operations for surface ventilation fans, UG pumping, compressed air and mining equipment.

Table 18-1: Power Demand

Description	Units	Value
Process plant	MW	2.9
Underground	MW	2.0

Three Maud Creek mine site power supply options were considered (all at a preliminary level only):

Mains power

Site generated power (diesel)

Site generated power (gas).

It was assumed that whichever option was selected as the preferred base case, it would be provided by a third party.

The town of Katherine is linked by a 132 kV transmission line to the Darwin-Katherine Interconnected System which supplies regulated electricity to the region. The transmission and distribution of power in the Northern Territory is the responsibility of the government owned Power and Water Corporation (PWC).

The first option is for electrical power provided via a dedicated 22 kV overhead (O/H) transmission line from the Katherine Power Station (gas fired) to a substation adjacent to the Maud Creek mine site. A preliminary route and capital contribution estimates for the proposed 22 kV transmission line have been provided to SRK by PWC based on a 2007 study. The proposed layout is shown in Figure 18-2. While the direct line distance from Katherine to Maud Creek is ~ 20 km, the distance of this route is approximately 40.5 km. This is as a result of the Power Station being located on the west side of Katherine and a new powerline corridor being run to the south of the town, following the site access road. The route is shown in Figure 18-1. The previous PWC study route was approximately 32 km long, but the currently available easement is now further south that the proposed 2007 route. Much of the infrastructure already exists along the proposed line, which should improve constructability and the approval requirements.

The estimated capital cost provided for the construction of the transmission line is AUD5.7M. In accordance with PWC' s power networks policy, the capital contribution would be approximately AUD3.6M and a prudential requirement (bank guarantee) of AUD2.1M. In a standalone mine and processing facility, this option becomes attractive but with the base case assuming processing at Union Reefs, the capital cost requirement of this option is high for the 2.0MW requirement of the mine only and is not considered in further detail.

On site power generation using gas fired generator sets was investigated. The budget costs to install a 2” polyurethane/carbon fibre gas pipe from the Stuart Highway to site was estimated by PWC as AUD5-6.0M, which is higher than the capital contribution associated with the transmission lines and for the same reason is not considered in further detail.

The preferred option is to have diesel generator sets provided, owned and operated by an independent power provider (IPP), to supply the electrical power required by the mine. This is a low capital cost option that suits the power demand and relatively short LoM. The overall charge for power generation at Maud Creek is AUD0.2875/kWh.

The power supply at the Union Reefs Processing Plant has an existing connection to the Northern Territory power grid. Network and electricity supply tariffs have been provided at an average unit power cost of AUD0.2177/kWh.

A more detailed assessment of onsite power generation will be undertaken in the next phase of the pre-feasibility study.

During periods of power outages, a ‘black start’ diesel generating unit will be used to provide sufficient power for emergency lighting, mine egress and to operate other critical equipment drives. Back up diesel power for critical drives is already available at Union Reefs.

The 11 kV supply will be fed underground via a service hole to a substation that steps the voltage down to 1000 volts and the power is reticulated to the working areas via cable and distribution boxes.

On the surface, power will be distributed to substations throughout the plant and stepped down to 415 V for use by mechanical equipment components. Distribution between the substations and the processing facilities will be undertaken in pipe racks using cable trays.

At the next level of design, a more detailed scheme for the distribution of power will be developed to obtain market pricing on substations, switch rooms and other electrical infrastructure.

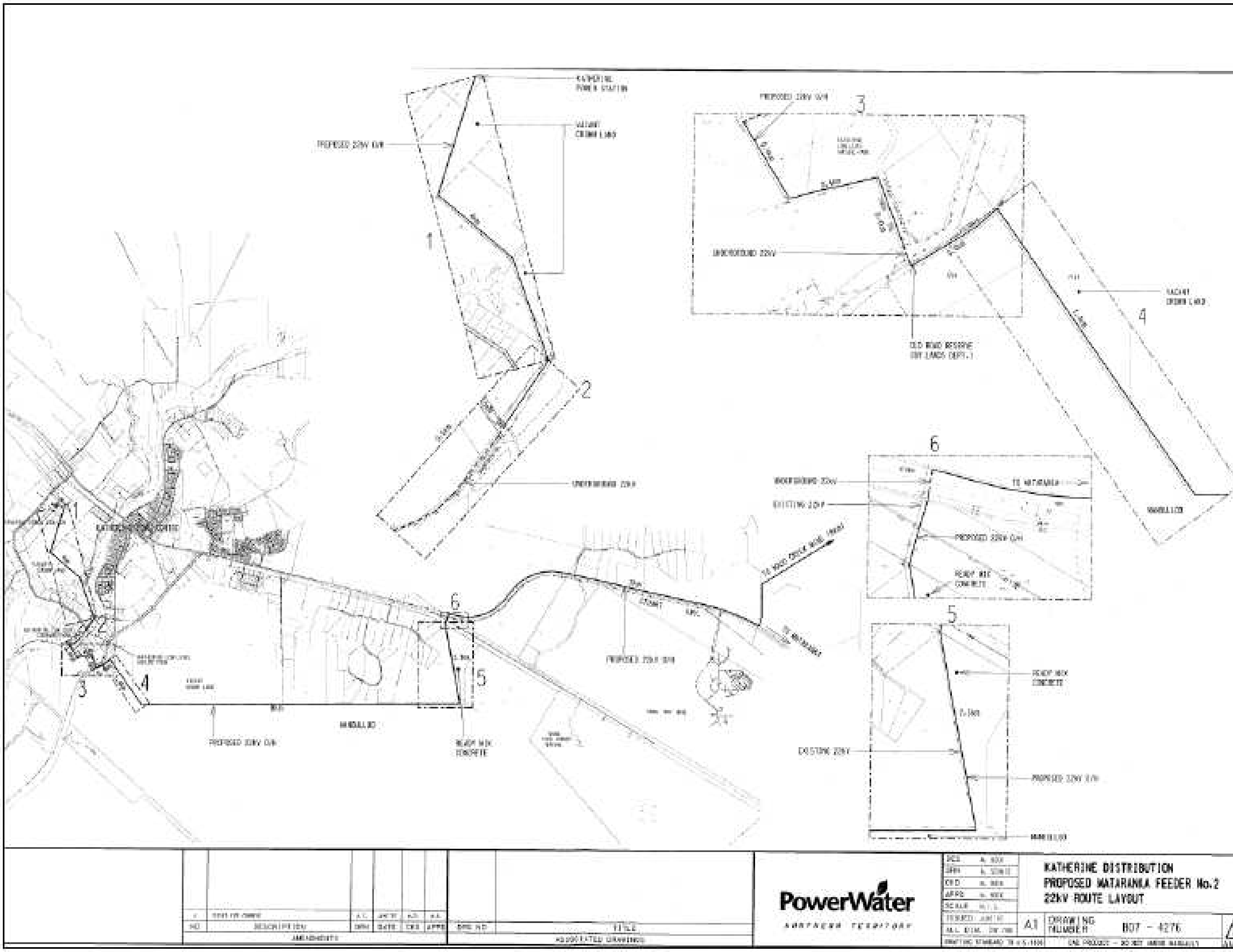


Figure 18-2: Proposed Power distribution

18.4 Water Supply

The raw water supply sources for the site include the:

- Groundwater seepage into the mine pit; and
- Potential runoff over catchments associated with the mine pit, waste rock dump, tailings facility infrastructure area, and sedimentation ponds.

The total catchment area reporting to the Maud Creek site of 25.74 km^2 was divided into three major sub-catchments, as shown in Figure 18-12. The areas of each sub-catchment are outlined in Table 18-2.

Water management of the site will need to be incorporated as a key factor in any future plans, to keep clean water clean and direct the contact water to appropriate containment system.

Table 18-2: Areal extent of catchment areas

ID	Area (km^2)
1	0.616
2	1.337
3	23.786

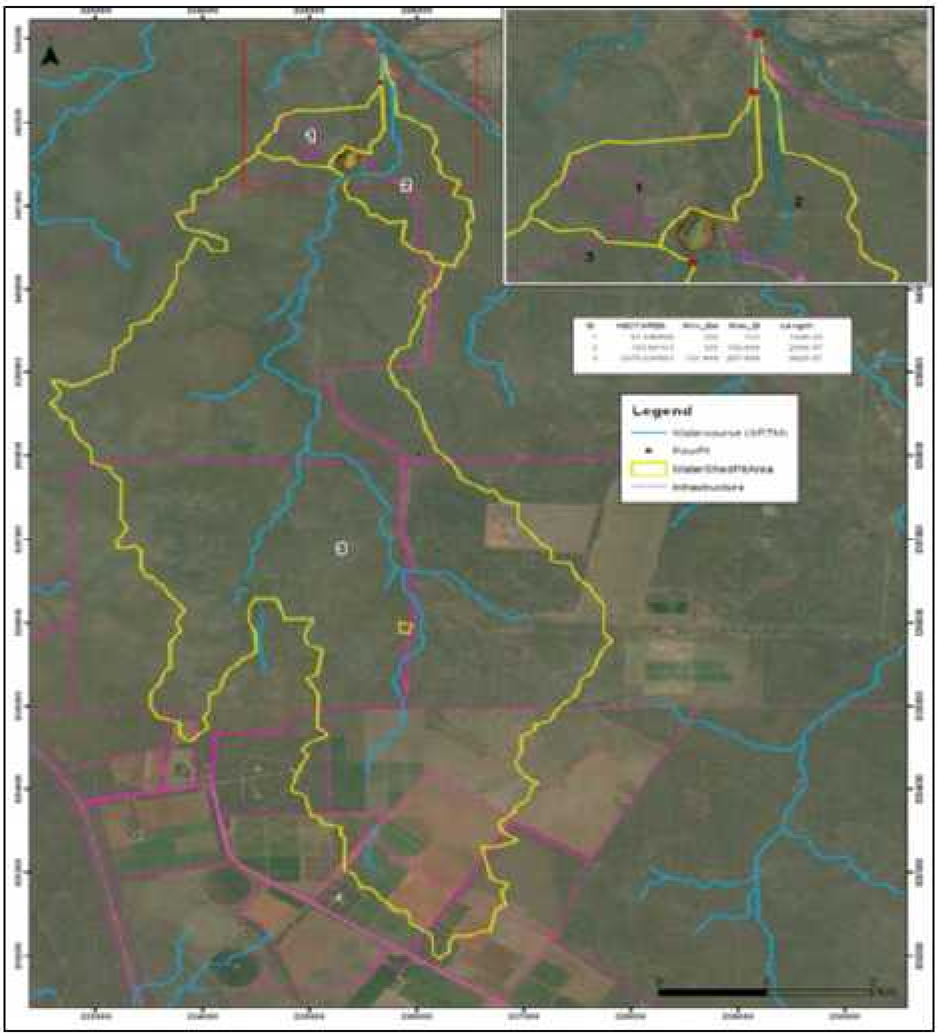


Figure 18-3: Catchment areas in Maud Creek site

18.5 Tailings Storage

The proposed processing route via the Processing Plant at Union Reefs, as such a Tailing Storage facility is not be required as part of the infrastructure requirements at Maud Creek.

The tailings disposal strategy for the Union Reefs process plant is to pump the tailings slurry to the Crosscourse pit tailings facility, with process waster recycled back to the Union Reefs Plant.

18.6 Surface Water Management - Hydrology

The primary objective for the surface water management of the Maud Creek project is to keep clean water clean and direct the contact water to appropriate containment systems. A number of water management strategies have been proposed, mainly utilising diversion and collection structures. The below-ground (Hydrogeology) aspects are discussed in Section 16.5.

The surface water management system for the Maud Creek project consists of the following elements:

- Flood protection bund to the east of the pit;
- Gold Creek diversion channel to the east of the pit;
- Channel to collect the contact water running off the flanks of the waste rock dump (WRD) which will report to the water pond;
- Water pond, receiving water from pit dewatering and surface runoff from WRD to feed the water treatment plant or to evaporate; and
- Emergency spillway to discharge contact water from the water pond into the environment in case of emergency.

The configuration of the water management system is shown in Figure 18-3.

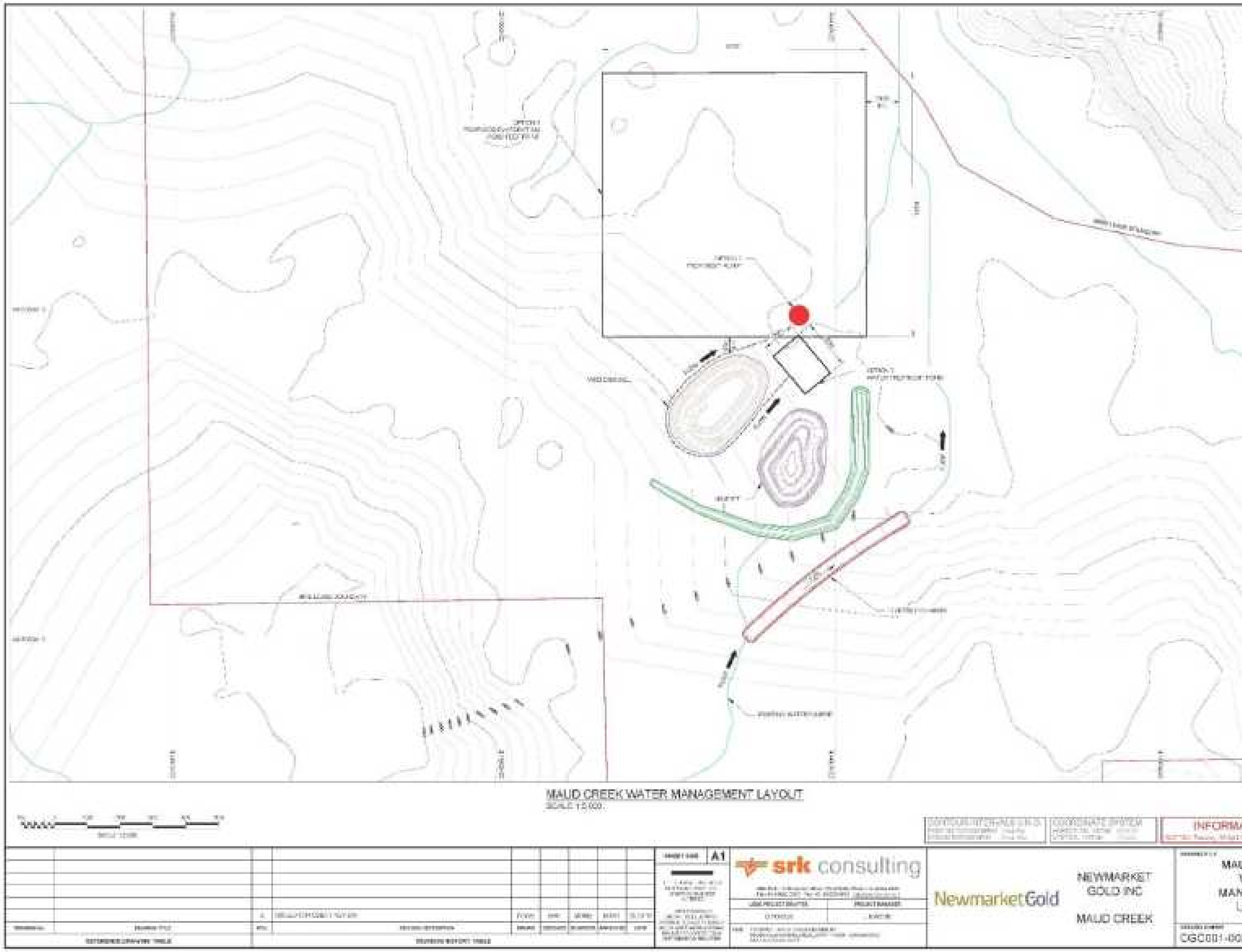


Figure 18-4: Water management configuration

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18.6.1 Peak flow calculation

Peak flows represent the highest possible flow that a given catchment, channel or other feature can experience for a given storm/ precipitation event. The peak flows for the channels were calculated using the Australian Rainfall and Runoff (AAR): A Guide to Flood Estimation. The relevant design criteria are shown in Table 18-3.

Table 18-3: Peak flow calculation method extracted from ARR

Parameter	Value
Design Average Recurrence Interval (ARI)	1 in 100 years
Region Definition	Northern Territory
Method	Rational Method
Concentration time (tc) estimate	$t_c = 58\ L/\ (A10^{0.1}\ Se^{0.20})$
Runoff Coefficient (C2) estimate	variable with terrain slope
Peak Flow (QY)	$Q_Y = 0.278C_2\ (\ CY/C2)1_{tc}A$

Catchment characteristics are used to estimate concentration times and the runoff coefficient, C2.

The intensity is determined by interpolating the Intensity, Duration and Frequency (IDF) data in Table 18-4 for the storm duration equal to the catchment' s concentration time.

Table 18-4: Intensities (mm/h) for various durations and return periods for Maud Creek

Duration		Average Storm Recurrence Interval (Years)								
hours	**min**	**1**	**2**	**5**	**10**	**20**	**50**	**100**	**500***	**1000***
0.08	5	110.0	141.0	178.0	200.0	230.0	271.0	303.0	365.8	394.4
0.10	6	103.0	132.0	166.0	187.0	215.0	253.0	283.0	341.5	368.2
0.17	10	85.6	110.0	138.0	155.0	178.0	209.0	233.0	281.4	303.2
0.33	20	65.8	84.1	105.0	117.0	134.0	157.0	174.0	209.9	225.9
0.50	30	54.7	69.9	86.6	96.7	111.0	129.0	144.0	173.1	186.2
1.00	60	37.0	47.2	58.5	65.2	74.6	87.0	96.7	116.3	125.1
2	120	22.8	29.1	36.1	40.3	46.2	53.9	60.0	72.2	77.7
3	180	16.6	21.3	26.5	29.6	33.9	39.7	44.1	53.2	57.2
6	360	9.6	12.2	15.3	17.2	19.7	23.2	25.8	31.1	33.6
12	720	5.6	7.2	9.2	10.3	11.9	14.1	15.8	19.1	20.6
24	1440	3.5	4.5	5.8	6.7	7.7	9.2	10.4	12.6	13.7
48	2880	2.2	2.9	3.8	4.4	5.2	6.3	7.1	8.7	9.4
72	4320	1.6	2.1	2.8	3.3	3.9	4.7	5.4	6.6	7.2

*Extrapolated values

In accordance with recommendations from AAR, peak flows were calculated using the Rational Method. This method is based on a simplified representation of the law of conservation of mass and the hypothesis that the flow rate in a catchment is directly proportional to the size of the contributing area and the rainfall intensity, with the latter a function of the return period.

The peak flows have been calculated for ARIs varying from 1 to 1000 years.

18.6.2 Gold Creek diversion bund and channel

A flood assessment has been undertaken to determine sizing requirements for a diversion bund and channel to divert storm flows around the pit. The two-dimensional flood routing software package FLO-2D was selected as the preferred software for the assessment because of its ability to simulate unconfined overland flow.

Model topography was sourced from the Shuttle Radar Topography Mission (SRTM) at a grid spacing of 30 m. The remaining model inputs are summarised in Table 18-5.

Table 18-5: Model inputs

Parameter	Value	Source / Comment
Design flood event	1-in-1000 year ARI	Assumed
Peak flow (m^3/s)	679.2	Calculated based on ARR recommendations
Time of concentration (hrs)	4.6	Calculated based on ARR recommendations
Manning’ s n - watercourses and overbanks	0.06	Average values sourced from Hardcastle & Richards flood study 1998
Manning’ s n - diversion channel	0.04	
Average diversion channel depth (m)	4	Hardcastle & Richards flood study 1998

The model was run at a grid spacing of 30 m to be consistent with the level of accuracy of the topographic information. The results are shown in Figure 18-4 and indicate that a minimum 9 m high bund is required to prevent flows associated with the 1-in-1000 year ARI event from encroaching the pit.

The diversion channel will have 4 m depth, 30 m wide and follow the natural slope. It will collect water from the catchment upstream of the pit. It will run south east of the pit and then discharge into the natural catchment.

A channel network was conceptually located to collect the contact water running off the flanks of the waste rock dump (WRD) which will finally report to the water pond.

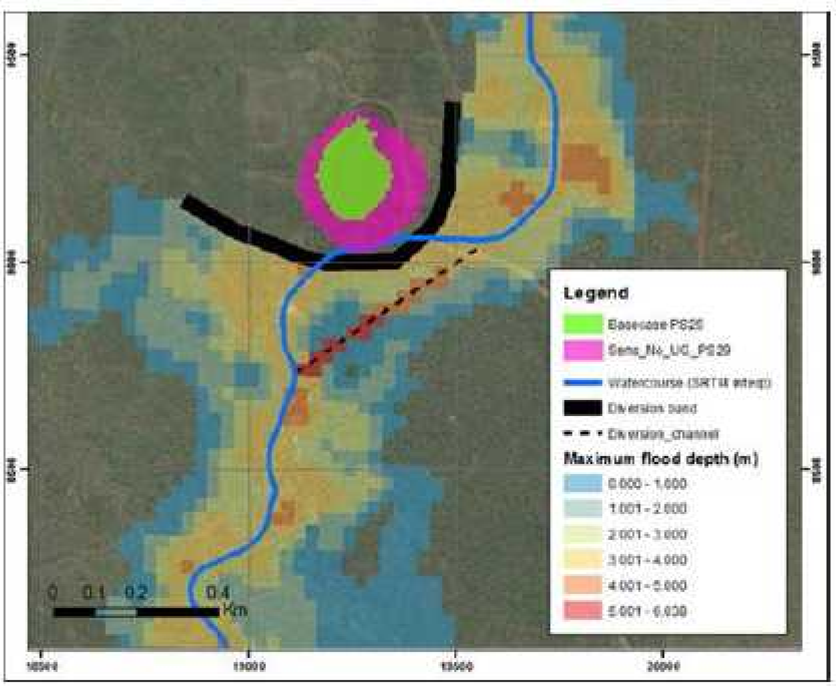

Figure 18-5: Flood routing results

18.6.3 Water pond

Two potential water pond sizes have been evaluated for the site as follows:

Option 1 considers an evaporation pond, large enough to contain and evaporate the water coming from the pit and also the runoff from the WRD.

Option 2 considers the installation of a water treatment plant with capacity of 3,200 m^3/d and a water pond to contain the excess water. Water from the pit and also runoff from the WRD will be directed to the water pond and from there send to the treatment plant.

Characteristics of the two water pond options are shown in Table 18-6 and their locations are shown in SRK Drawing CGC001-020. A spillway designed for closure will be constructed in the water pond. It will be a monitoring point for water quality and quantity.

The pond area required to rely on evaporation (Option 1) is almost forty-three times larger than considering a treatment plant (Option 2). It is recommended to explore both options when more details regarding water quality are available.

Table 18-6: Water ponds capacities

Water pond	Treatment Capacity (m^3/d)	Pond Area (m^2)	Pond Depth (m)	Pond Volume (m^3)	Freeboard (m)
Option 1	0	640,000	4.0	2,560,000	1.06
Option 2	3,200	15,000	2.5	37,500	0.68

18.6.4 Water demand and supply assessment

Two water balance models were developed to assess the mine water availability, supply and containment based on the water management system described in Section 18.8.

A monthly water balance was developed considering an evaporation pond (water pond Option 1) and a daily water balance was developed to consider the effect of the water treatment while evaluating a smaller pond. The models simulate precipitation, evaporation, pit-dewatering and water storages, for the two water pond options during the mine life (10 years). The raw water supply sources for the site include the following:

Groundwater seepage into the open pit; and

Direct precipitation over the water pond; and

Potential runoff over catchments associated with the open pit and waste rock dump.

The only water demand considered in the system is dust suppression at an estimated rate of 240m^3/d.

The daily model takes into account the ability to release water from the water treatment plant to the natural streams after checking water quality standards are met. Key findings of the monthly water balance for Option 1 are:

The water demand is met all the time during operations, Figure 18-5, where total inflows are larger than total outflows;

There is excess water in the system to be stored and accumulated while evaporation occurs;

The maximum accumulated water volume over the ten years of mine life is approximately 2Mm^3, Figure 18-6;

The total inflows are greatly influenced by the pit dewatering which decreases yearly; and

Rainfall and runoff is not a significant or reliable source of water for mine operations. This is due to the climatic characteristics of the site, which has episodic rainfall and high evaporation.

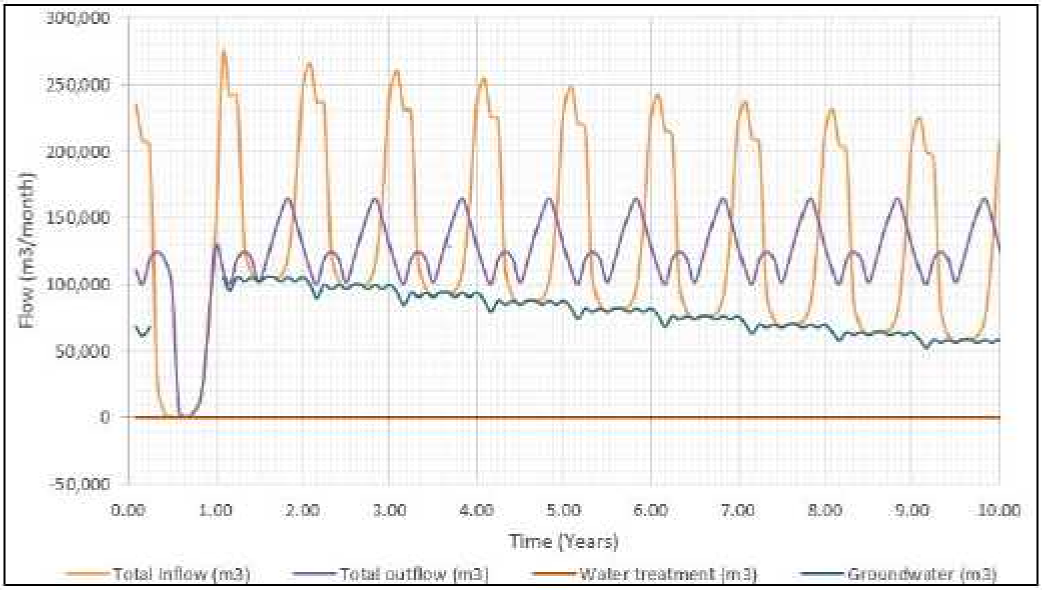

Figure 18-6: Total monthly inflows and outflows - Option 1 system

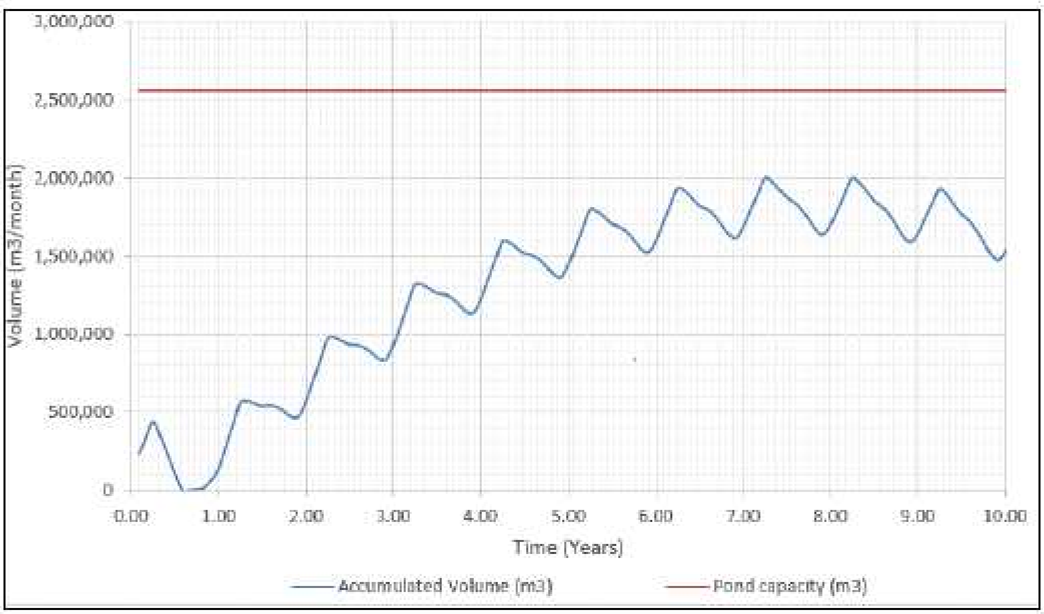

Figure 18-7: Monthly water accumulated volumes and pond capacity - Option 1

Key findings of the daily water balance for Option 2 are:

The water demand is met all the time during operations, Figure 18-7 where total inflows are larger than dust suppresion requirement.

The total inflows are greatly influenced by the pit dewatering which decreases yearly.

Total inflows exceed total outflows during year two and three only.

There is excess water in the system to be stored particularly during years two and three.

The maximum accumulated water volume after treatment over the ten years of mine life is approximately 27,300m^3, Figure 18-8.

Rainfall and runoff is not a significant or reliable source of water for mine operations. This is due to the climatic characteristics of the site, which has episodic rainfall and high evaporation.

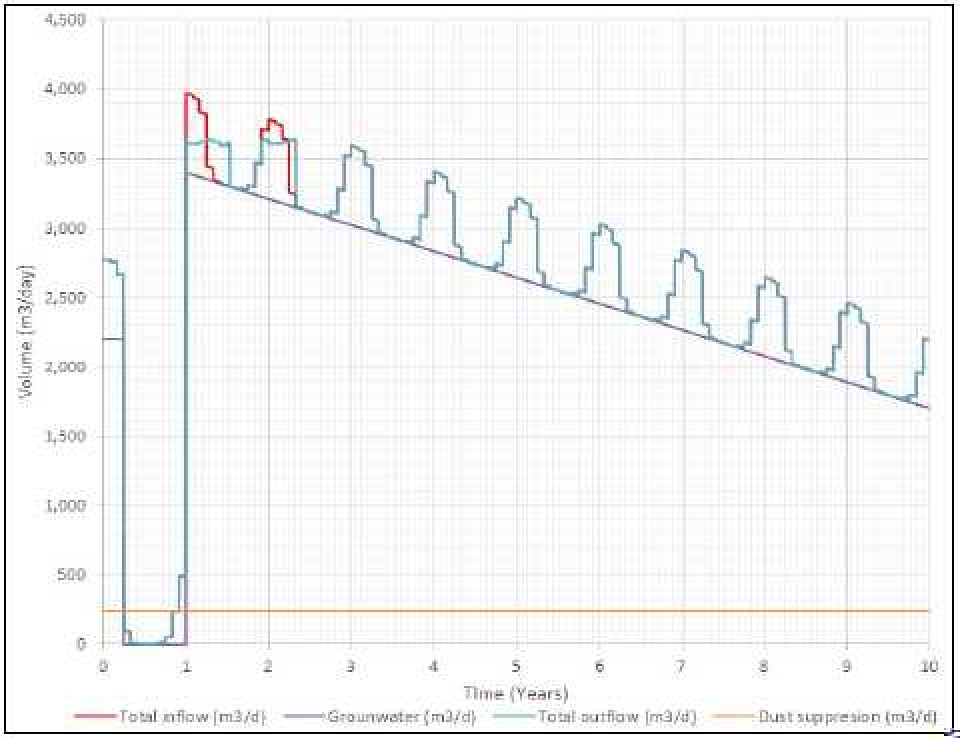

Figure 18-8: Total inflows and outflows - Option 2 system

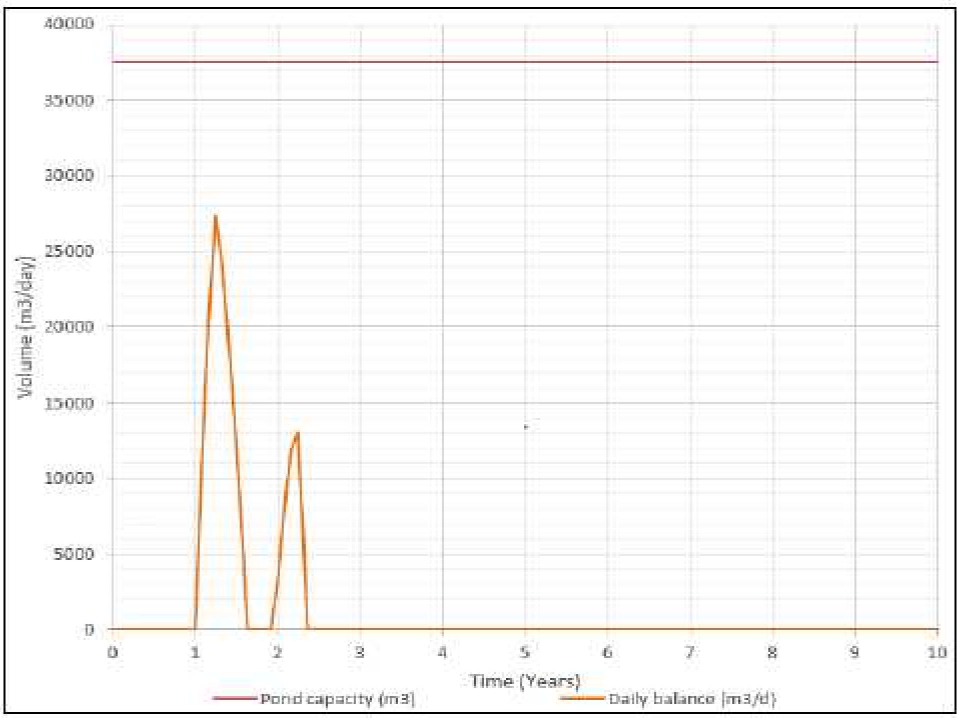

Figure 18-9: Water accumulated volume and pond capacity - Option 2

18.6.5 Preliminary cost estimate

A cost estimate have been prepared based on the preliminary schedule of quantities developed for major cost items for both Option 1 (Table 18-7) and Option 2 (Table 18-8). Average unit costs were taken from recent tenders received from WA contractors for construction work.

SRK has assumed a cost for a water treatment plant to be AUD5M.

Table 18-7: Preliminary cost estimate for surface water Option 1

ITEM	DESCRIPTION	UNIT	QUANTITY	RATE (AUD)	COST (AUD)
A	**Preliminary and General**				**5,897,374**
A.1	Mobilization, time related running costs, and de-mobilization; assumed 25% of the sum of all other costs	Lump Sum	1	5,897,374	5,897,374
B	**Earthworks**				**17,141,439**
B.1	**Clear / Grubbing / Topsoil Stripping**				
B.1.1	Clear, grub and rip pond footprint (assumed 150mm depth)	m²	640,000	1	422,400
B.1.2	Strip topsoil from pond footprint (about 250mm depth)	m³	160,000	4	641,600
B.1.3	Clear, grub and rip channel footprint (assumed 150mm depth)	m²	14,625	1	9,653
B.1.4	Strip topsoil from channel footprint (about 250mm depth)	m³	10,000	4	40,100
B.1.5	Clear, grub and rip Gold Creek diversion footprint (assumed 150mm depth)	m²	28,150	1	18,579
B.1.6	Strip topsoil from Gold creek diversion footprint (about 250mm depth)	m³	10,000	4	40,100
B.1.7	Clear, grub and rip bund protection footprint (assumed 150mm depth)	m²	41,575	1	27,440
B.1.8	Strip topsoil from bund protection footprint (about 250mm depth)	m³	20,000	4	80,200
B.2	**Excavation**				
B.2.1*	Excavate in situ material (pond) and either direct place into the diversion embankment or stockpile for re-use.	m³	3,072,000	5	14,784,000
B.2.2*	Excavate in situ material (channel) and either direct place into the diversion embankment or stockpile for re-use.	m³	14,040	5	67,568
B.2.3*	Excavate in situ material (diversion) and either direct place into the diversion embankment or stockpile for re-use.	m³	110,000	5	529,375
B.3	**Compacted Earthworks**				
B.3.1	Spread, moisture condition, and compact to specification, excavated in situ fill material to form bund embankment	m³	172,350	3	480,426
C	**Lining**				**6,448,056**
C.1	Supply and install HDPE liner to prepared base of pond	m²	640,000	10	6,304,000
C.2	Supply and install HDPE liner to prepared base of channel	m²	14,625	10	144,056
D	**Treatment plant**				
D.1	Supply and install treatment plant	Lump Sum	0		NA
TOTAL					**29,486,869**

*Includes 20% bulking factor

Table 18-8: Preliminary cost estimate for surface water Option 2

ITEM	DESCRIPTION	UNIT	QUANTITY	RATE ( )	COST ( )
A	**Preliminary and General**				**462,952**
A.1	Mobilization, time related running costs, and de-mobilization Assumed 25% of the sum of all other costs	Lump Sum	1	462,952	462,952
B	**Earthworks**				**1,560,002**
B.1	**Clear / Grubbing / Topsoil Stripping**				
B.1.1	Clear, grub and rip pond footprint (assumed 150mm depth)	m²	15,000	1	9,900
B.1.2	Strip topsoil from pond footprint (about 250mm depth)	m³	10,000	4	40,100
B.1.3	Clear, grub and rip channel footprint (assumed 150mm depth)	m²	14,625	1	9,653
B.1.4	Strip topsoil from channel footprint (about 250mm depth)	m³	10,000	4	40,100
B.1.5	Clear, grub and rip Gold Creek diversion footprint (assumed 150mm depth)	m²	28,150	1	18,579
B.1.6	Strip topsoil from Gold creek diversion footprint (about 250mm depth)	m³	10,000	4	40,100
B.1.7	Clear, grub and rip bund protection footprint (assumed 150mm depth)	m²	41,575	1	27,440
B.1.8	Strip topsoil from bund protection footprint (about 250mm depth)	m³	20,000	4	80,200
B.2	**Excavation**				
B.2.1*	Excavate in situ material (pond) and either direct place into the diversion embankment or stockpile for re-use.	m³	45,000	5	216,563
B.2.2*	Excavate in situ material (channel) and either direct place into the diversion embankment or stockpile for re-use.	m³	14,040	5	67,568
B.2.3*	Excavate in situ material (diversion) and either direct place into the diversion embankment or stockpile for re-use.	m³	110,000	5	529,375
B.3	**Compacted Earthworks**				
B.3.1	Spread, moisture condition, and compact to specification, excavated in situ fill material to form bund embankment	m³	172,350	3	480,426
C	**Lining**				**291,806**
C.1	Supply and install HDPE liner to prepared base of pond	m²	15,000	10	147,750
C.2	Supply and install HDPE liner to prepared base of channel	m²	14,625	10	144,056
D	**Treatment plant**				
D.1	Supply and install treatment plant	Lump Sum	1	5,000,000	5,000,000
TOTAL					**7,314,760**

*Includes 20% bulking factor

18.6.6 Conclusions

Water management of the site has been incorporated as a key factor, to keep clean water clean and direct the contact water to appropriate containment systems. A preliminary analysis has been completed to prevent flows associated with the 1-in-1000 year ARI event from encroaching the open pit.

The designed infrastructure includes a flood protection bund to the east of the pit, a diversion channel to maintain Gold Creek to the east of the pit has been conceptually designed and a dedicated water pond for the excess water.

Based on the available data, the preliminary water balance indicates the water demand is met all the time during operations. There is excess water in the system to be treated and managed on-site.

18.7 Surface Haulage

A suitable access road connecting the Maud Creek mine site to the Stuart Highway is required to facilitate the delivery of ROM material to the Union Reefs Processing Plant. Three route options are illustrated below in Figure 18-9.

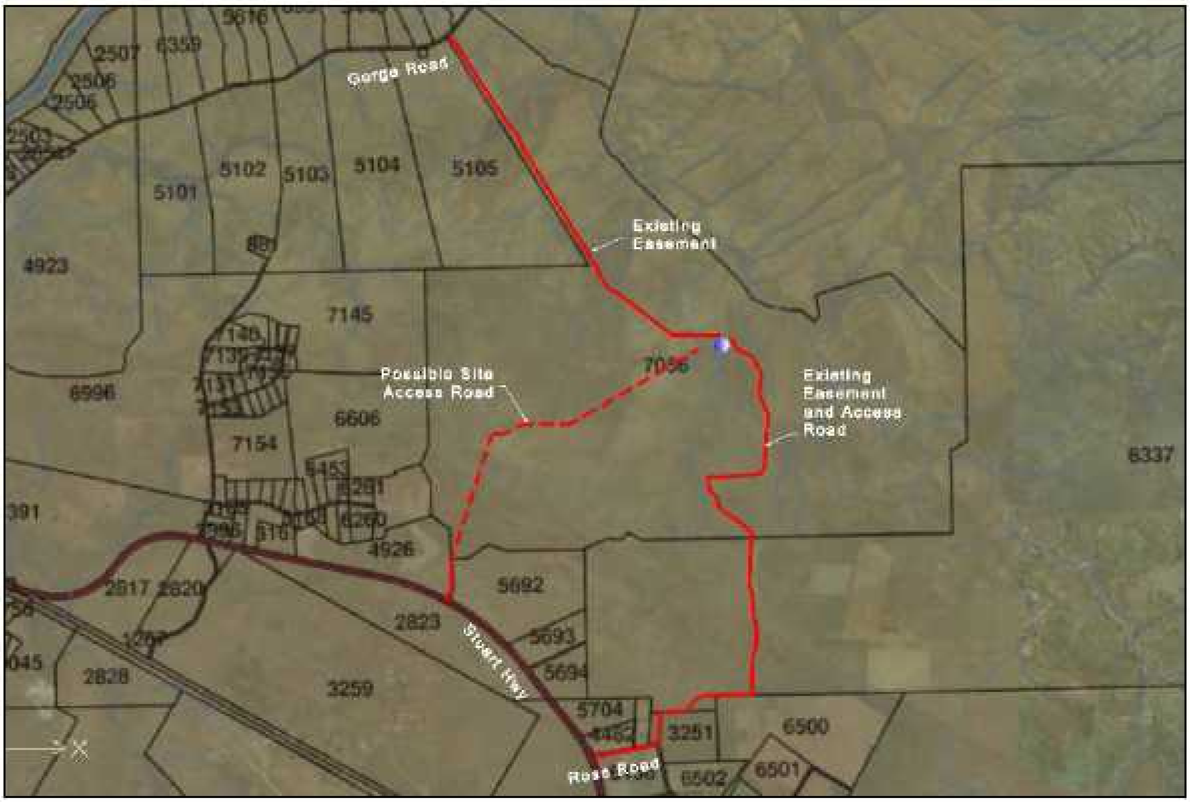

Figure 18-10: Maud Creek Site Access Roads

The use of rail via the Katherine Freight Terminal was not considered at this phase of study because of the relatively short haulage distance, the additional rehandle costs, the capital cost of a new siding and laydown area at Union Reef, the required approvals and access issues and the short LoM.

The existing unsealed access road to the Maud Creek site into Katherine, heading south from the mine was used during previous mine operations to haul material from site. The access road is linked to the Stuart Highway, a major sealed arterial route, by Ross Road, a sealed public road which is regularly used by horticultural landholders in the area. The access road crossed Gold Creek just south of the existing mine pit and during the wet season, the road is often impassable and experiences significant damage.

To facilitate the transport of material from site, this existing access track would need to be upgraded and have a crossing installed over Gold Creek to enable transport operations to continue during the wet season both into and from site. This route has been approved by the Northern Territory Government as a Right of Way and General Service easement.

A second approved Right of Way and General Service easement runs north from the Mine to Gorge Road. This provides the site with a second access point to Katherine.

A possible new access route runs south west from the mine pit and joins a short unnamed gazetted road before exiting onto the Stuart Highway. The new route is approximately 4.5 km shorter than the existing road and remains exclusively on Lot 4192. Although this route may represent some savings in overall haulage costs, it is not an existing easement. Approvals will be required and significant heritage and environmental concerns would need to be addressed and therefore has been disregarded at this time as a viable alternative transportation route.

The proposed option is to upgrade the southern access track that connects to Ross Road.

Once on the Stuart Highway, the proposed haulage route heads North through the township of Katherine to the Ping Que Road turnoff. The proposed haulage route is presented in Figure 18-11. At the next level of design, a more detailed road and service provision can be made with further detailed information regarding the flood plans near the Gold Creek.

The budget cost to upgrade the current track with a provision for flood protection is AUD2.2M. An additional 20% contingency has been applied due to the level of energy undertaken. This cost is in included in the capital cost estimate in Section 21.

Figure 18-11: Haulage route from Maud Creek Mine to Union Reefs Processing Plant

18.7.1 Haulage Operations and Mine Traffic

It is anticipated that the mine haulage operations will utilise quad semi-trailers, which will involve travel from Maud Creek to Union Reefs (loaded) and back again (empty) on a daily basis; therefore, the mining operations will require approximately 26 road train movements per day (13 full and 13 empty). A summary of the haulage logistics is shown in Table 18-9.

A typical loaded quad semitrailer from Maud Creek mine site will have 22 axles, and a total/payload weight of about 160/ 110 tonnes respectively and complies with the Department of Transports maximum mass regulations. Confirmation of allowable number of trailers and load through town centres is required at the next phase of the pre-feasibility study. Any reduction will require a corresponding increase in truck movements each day.

Table 18-9: Haulage Logistics

Description	Units	Value
Annual Throughput	t/y	500,000
Number of shifts per day	No	1
Shift duration	h	12
Operating hours per day	h/d	10.8
Operating days per year	d/y	360
Operating hours per year	h/y	3,888

Description	Units	Value
Daily capacity required (average)	t/d	1,389
Truck payload (quad)	t/truck	110
Return trips per day	trips/d	13
Return trip distance	km/trip	288
Return trip duration	h/trip	4.73
Return trips per truck per day	trips/d/truck	2.28
Trucks required	**No**	**6**
Truck separation time	min	55

Based on 227.8 km @ 90km/hr, 60km @ 50km/hr, 1-hour loading/unloading per trip

Site supplies will be transported to the Maud Creek mine site from Darwin, Pine Creek area, Katherine or from South and Western Australia as required.

Staff commuting to the mine site will mainly be travelling from Katherine, via cars and minibuses, with commuting occurring across a 24-hour period throughout the life of the mine. Approximately 15 small vehicles including one or two minibuses are likely to be used daily to transport employees to site.

18.7.2 Impacts on Traffic

A summary of the existing traffic and the traffic volumes that are expected from the proposed mining activities including various types of vehicles on Stuart Highway are presented in Table 18-10.

The traffic volumes are based on the Department of Transports published traffic data at various locations along the proposed haul route. It can be seen from the above traffic volumes that there is a higher volume of traffic north of Katherine than south of it. This is due to the additional traffic between Katherine and Darwin and also the traffic travelling through Katherine using the Victoria Highway.

It is proposed that haulage from the site and traffic to the site will occur only during daylight hours. This would nominally be from 6:00AM to 6:00PM. This will occur throughout the year during both the wet and dry seasons. The proposed quad road train and light vehicle movements have been combined with the 2014 traffic numbers in Table 18-10 to assess the impact on traffic numbers.

Table 18-10: Stuart Highway Traffic Counts and Classification

Description	Units	Vehicle Class					Total
		Short Vehicle	Short Towing Vehicle	2, 3, 4 Axle Trucks	3-6 Axle Articulated Vehicle	B Double, Double & Triple Road Trains	
		1	2	3-5	5-9	10-12	
Stuart Highway - 30 km south of Katherine							
2014 Traffic count figures	%	89.8				10.2	100
	No.	853				97	950
Proposed new traffic	No.	40				26	66
New total	No.	893				123	1016
Percent increase	%	4.7				26.9	6.9
Stuart Highway - 20 km north of Katherine							
2014 Traffic count	%	92.9				7.1	100

Description	Units	Vehicle Class					Total
		Short Vehicle	Towing Vehicle	2, 3, 4 Axle Trucks	3-6 Axle Articulated Vehicle	B Double, Double & Triple Road Trains	
		1	2	3-5	5-9	10-12	
figures	No.			1383		106	1489
Proposed new traffic	No.			10		26	36
New total	No.			1393		132	1525
Percent increase	%			0.7		24.6	2.4
Stuart Highway - 2km North of Kakadu Hwy (near Unions Reefs)							
2014 Traffic count figures	%			90.1		9.9	100
	No.			1033		113	1146
Proposed new traffic	No.			10		26	36
New total	No.			1043		139	1182
Percent increase	%			1.0		22.9	3.1
Stuart Highway - 100m South of Victoria Hwy within Katherine urban zone							
2014 Traffic count	No.						9179
Proposed new traffic	No.			40		26	66
New total	No.						9245
Percent increase	%						0.7

2014 Traffic count figures taken from the Department of Transports “ *Annual Traffic Report 2014”*

Overall the increase in traffic on the Stuart Highway resulting from the mining operations at Maud Creek would be 2.4 percent north of Katherine, 6.9 percent south of Katherine and 0.7 percent within the Katherine Township. A graphical representation of the increased traffic numbers is provided in Figure 18-11.

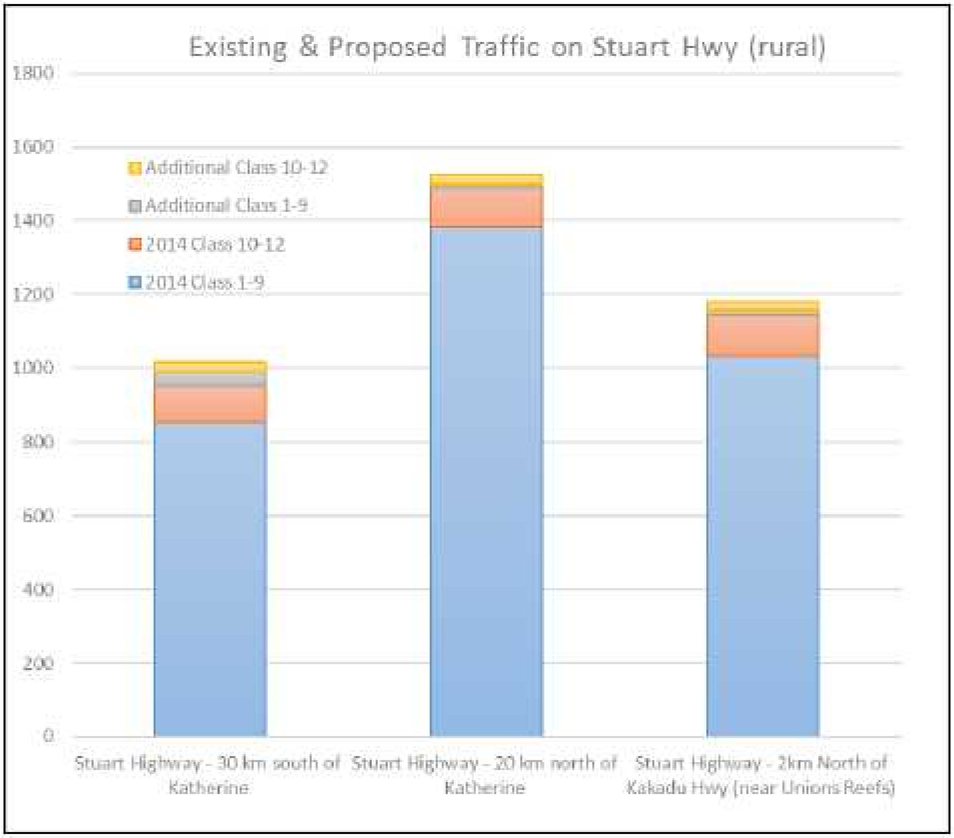

Figure 18-12: Impacts on traffic near Katherine and Union Reefs

This number of small vehicles represents less than a 5 percent increase in the existing traffic levels, which is considered minor and is not anticipated to pose a significant safety or traffic issue for other drivers in the area.

The 26 quad road train movements per day represents a 24.6 percent increase north of Katherine and a 26.9 percent south of Katherine. The major safety impacts of these road trains will be at the two intersections where the road trains turn onto or off the Stuart Highway and ensuring adequate overtaking opportunities for cars.

The Ross Road and Stuart Highway intersection currently has a left turning lane into Ross Road for unloaded road trains returning from the Union Reefs Processing Plant. The Ping Que Road and Stuart Highway intersection currently has an additional lane on the Stuart highway to allow traffic to move past a laden road train turning right into Ping Que Road. In future studies, a more detailed assessment of these intersections will be required to assess the adequacy and the level of safety the turning lanes provide.

Heavy vehicle route maps published by the Department of Transport show that the current Katherine road train route follows the Stuart Highway through the centre of township. As shown in Table 18-10, the percentage increase in traffic through the township is less than 1 percent. Speed limits through the town are set at 50 km/h to reduce safety risks to pedestrians and small vehicles.

In July 2015, the Northern Territory Government started the planning on a long-term heavy vehicle alternate route that bypasses the central business area of the township. The alternate heavy vehicle route study was proposed following a number of road safety issues in the main street, as it currently accommodates a mix of pedestrians, local and tourist traffic, and heavy vehicles. The alternate heavy vehicle route would dramatically reduce the impact of the Maud Creek haulage operations. Government and public consultation and approval will be a key element to the viability of processing of Maud Creek mineralization at the Union Reefs processing Plant.

18.8 Concentrate Transport

The flotation gold concentrate is loaded into 1.5 t capacity bulk bags, which are then loaded into shipping containers ready for transportation. The containers are loaded onto a road train that transports the concentrate by road from the Union Reefs Processing Plant to the Darwin Port. Railway transport has not been considered due to the need for a siding, capital cost, the relatively small volume of concentrate, short LoM and proximity and connectivity to Darwin via a major highway.

Costs were requested from Northline for the overland transport of the containers and have been included in the operating cost.

18.9 Site Facilities

Diesel Storage

Self-bunded diesel storage tank facilities of 80,000 - 100,000 litres will be located at the mine. This diesel storage caters for all underground and surface diesel needs as well as emergency black start fuel for back diesel generators for critical processing equipment.

Maintenance Facilities

A surface maintenance workshop facility will be located in the vicinity of the mine to service and maintain the underground fleet and associated mining equipment. All servicing and maintenance activities are undertaken on surface (i.e. none underground).

A maintenance/ boilermaker workshop will be located at the Processing Plant to assist with undertaking the required maintenance activities.

18.10 Housing and Land

The closest centre of population to the Maud Creek Gold Project is Katherine, which is a regional centre that enjoys excellent infrastructure, services and communications. The Airport at Katherine is serviced by Airnorth and has regular flights to and from Darwin.

Based on preliminary discussions with Newmarket Gold, it has been assumed that the workforce will consist of local/relocated residents, drive in/drive out from Darwin and other local towns such as Pine Creek and Adelaide River, and drive-in drive-out personnel to supplement the overall workforce in equal proportions.

Rented housing and hotel accommodation is available for mining personnel in Katherine. The Ibis Styles Hotel is located to the east of Katherine on the Stuart Highway and has approximately 100 rooms.

The 112 person Pine Creek camp is located approximately 120 km from the Maud Creek site and is currently in care and maintenance. The commute time to this camp makes it unsuitable for housing workers from the Maud Creek Mine.

The Cosmo accommodation village is used to house the Cosmo Underground Mine employees and contractors as well as the Union Reefs Processing Plant which is approximately 53 km south. This is also available for the very small increase in numbers required for processing and haulage contractors.

19 Market Studies and Contracts

19.1 Marketing

The third party toll treatment options assessment was based on indicative terms provided to Simulus Engineers for the sale of a gold concentrate to Shangdong Zhong Guo (China Gold Shandong) for processing at their Yantai Gold Smelter. Similar terms were provided by Baxville (Beijing) Minerals Trading Ltd.

MRI Trading AG has also been approached to provide indicative terms for the Maud Creek concentrate. They are facilitating the sale of KBL' s Mineral Hill gold concentrate. Dialogue continues but they have not yet provided final confirmation of pricing. Verbal confirmation of terms has been provided by KBL and has been used to benchmark the terms selected as the base case assumption.

Other inquiries have been issued to potential customers and traders. Indicative terms for the sale to another direct exporter of concentrate (i.e. to consolidate with their concentrate) who sell to Guoda Gold Co Ltd, or sale to an Australasian POX facility with additional capacity have been referenced from other similar gold concentrates projects and operations in Australia. Informal discussions have also been held with the owners of a number of Australian third party refractory gold operations for sale of a generic gold concentrate and to assess available capacity at those facilities but discussions have not been progressed to any great extent. It has been used mainly in the process of elimination of the bulk of these options. Further investigations into the remaining options can be undertaken at the next stage of study if deemed appropriate.

The sale of the gold concentrate to China remains the current base case. No formal concentrate discussions or sales contracts have been entered into at this stage of study.

Gravity gold will be recovered into gold doré on site and sold to established Australian refiners.

19.2 Contracts

No formal concentrate sales contracts are required at this stage of study, nor have any been arranged.

20 Environmental Studies, Permitting, and Social or Community Impact

20.1 Environment and Social Aspects and Impacts

20.1.1 Social and Economic Context

The Maud Creek Project lies within the Town of Katherine local government area (LGA 72200), which occupies an area of some 7417 km^2 (Figure 20-1) and within the broader Katherine region (336,674 km^{2,} Figure 20-2). The traditional owners of the land, the Jawoyn people, have occupied the Katherine region for thousands of years.

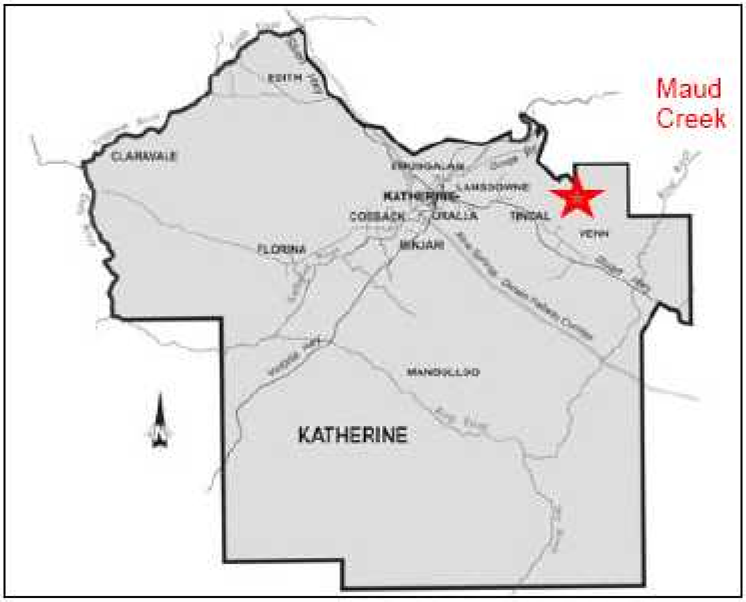

Figure 20-1: Katherine municipality local government area (ABS, 2011 census)

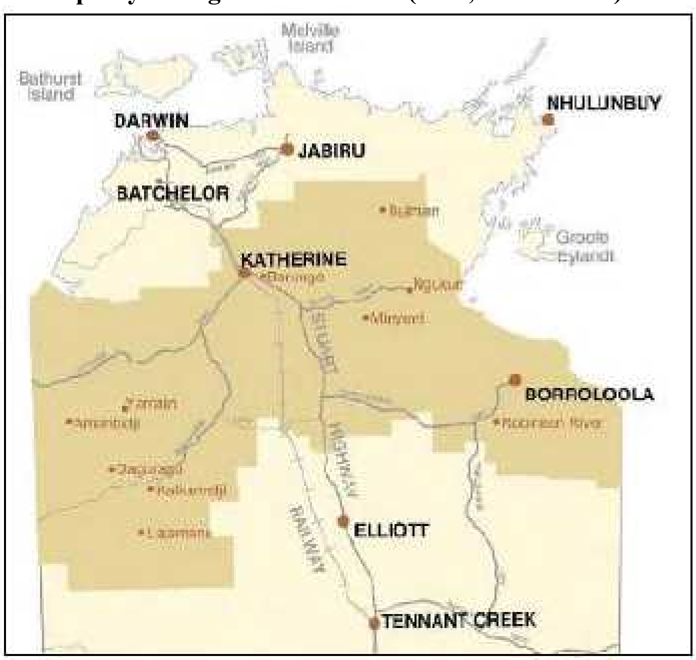

Figure 20-2: Katherine region (shown in darker tan)

Source: Katherine Land Use Plan, 2014

The town of Katherine is the fourth largest population centre in the Northern Territory, after Darwin, Palmerston and Alice Springs. The town was formally gazetted in 1926. At the 2011 census, the population of the Katherine municipality was estimated at slightly under 11,000 people, of whom about 24% identified as indigenous (Aboriginal or Torres Strait Islander). In younger age groups, the percentage of Aboriginal people is higher (Figure 20-3). Compared to the Australian population as a whole, the Katherine community is relatively young, with a median age of 31. The Katherine population is characterised by a high degree of mobility, with a relatively large proportion of the population changing place of residence (either within the region or interstate) between consecutive censuses.

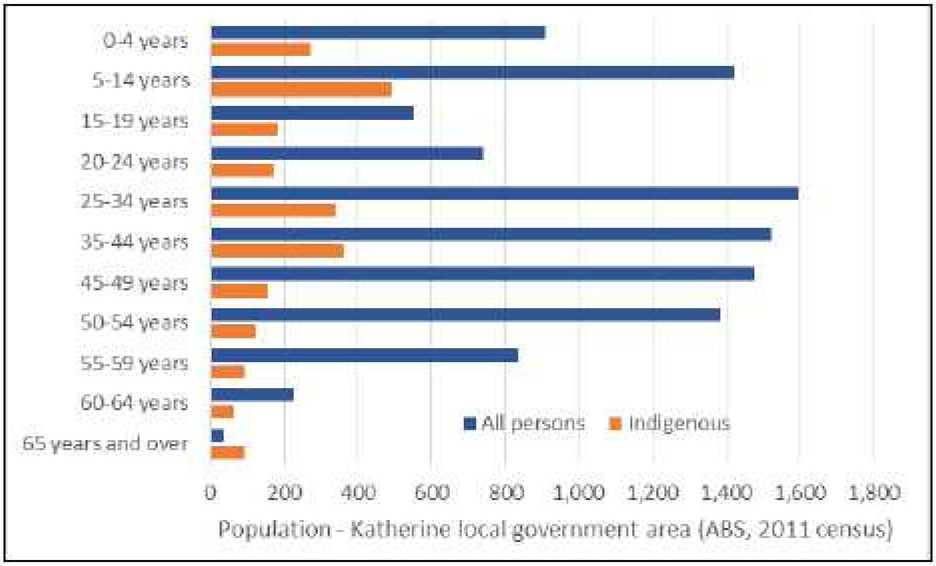

Figure 20-3: Katherine LGA population, by age group (2011 census)

Katherine is an important regional centre, providing government services (health, education, transport, communications and business development functions) to the wider region. A range of basic utilities, services and community infrastructure is available, including:

A 60-bed hospital; various community and private medical clinics;

A range of public and private education and training providers from pre- primary to university level; an airport (shared with the RAAF Base Tindal);

Police, fire and emergency services;

Water supply, sewerage and waste disposal facilities, and

Cultural, sport and outdoor leisure facilities, such as the Nitmiluk National Park.

The modern economy of the Katherine region has traditionally been dominated by the agricultural and pastoral sectors, but in the past few decades mining, public administration and safety (including defence), tourism and construction (largely related to major resource and infrastructure projects) have also been important contributors to the regional economy. Although mining has been by far the greatest contributor to gross regional product in the past few years (Figure 20-4), it is not a major employer in the Katherine municipality (Figure 20-5).

Unemployment and labour force participations rates in Katherine are, on average, similar to those in Australia as a whole, although unemployment among young people (those aged 24 years and below) is conspicuously higher than average unemployment rates in the general Australian community (Figure 20-6 and Figure 20-7). In 2011 about 41% of the Katherine population aged 15 and over had some type of post-secondary school qualification, mostly at Certificate or Diploma levels.

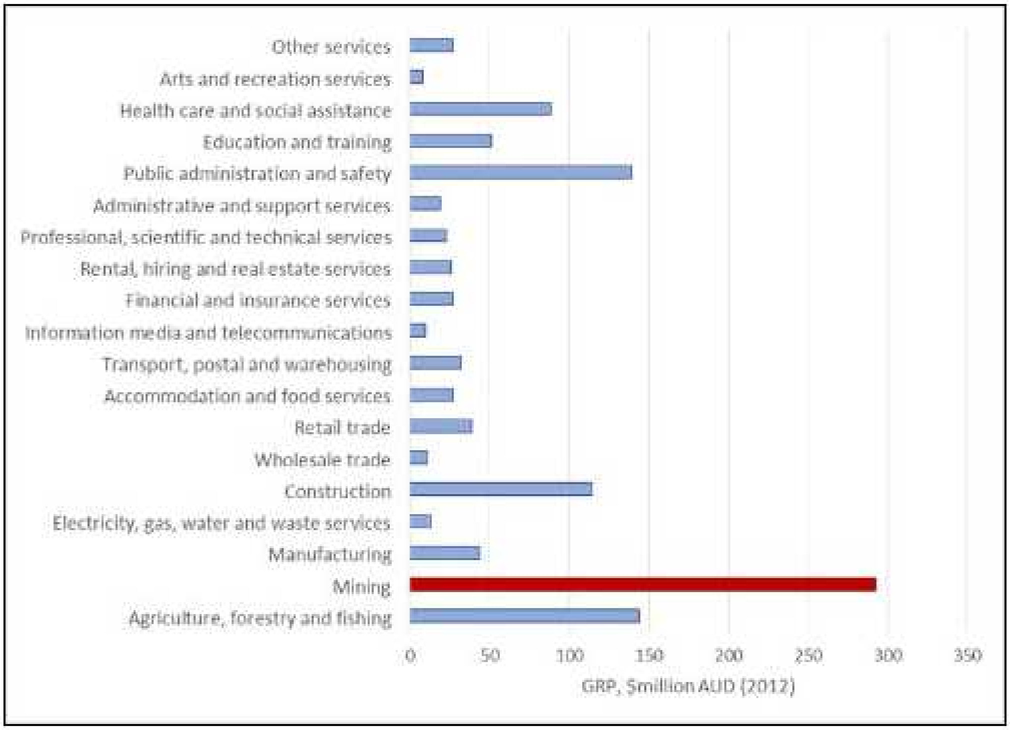

Figure 20-4: Gross regional product, by industry – Katherine NT (2012)

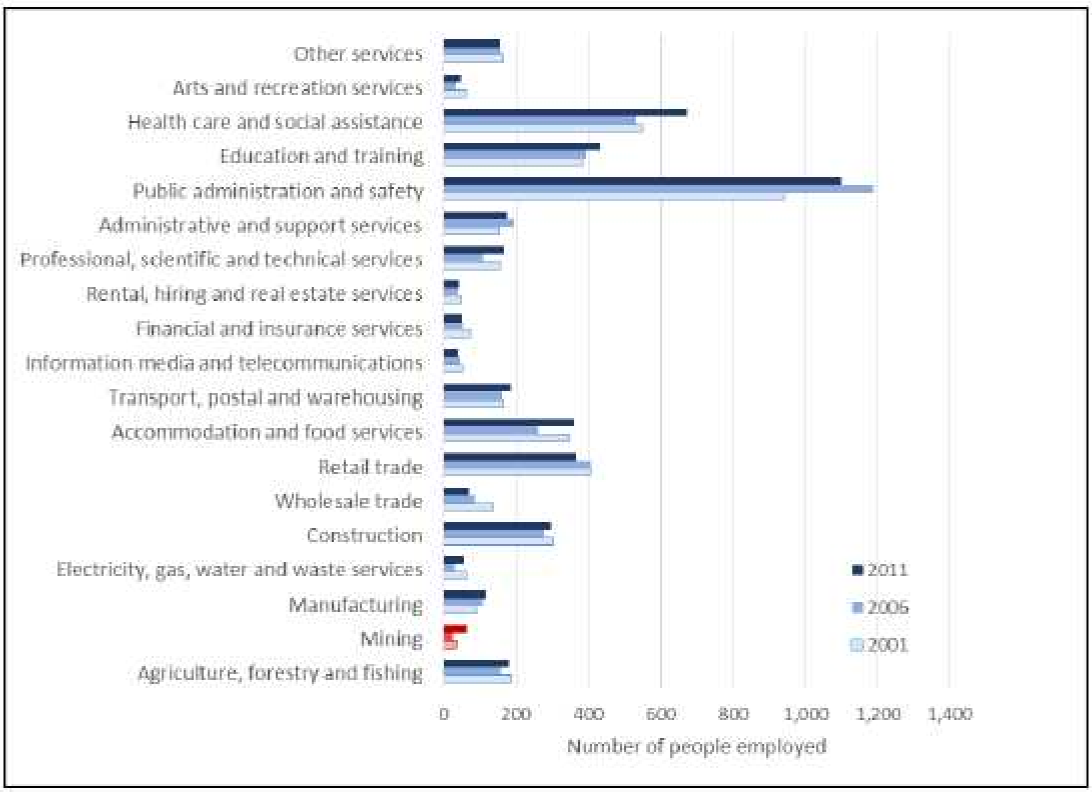

Figure 20-5: Employment by industry sector – Katherine LGA

Source: ABS data

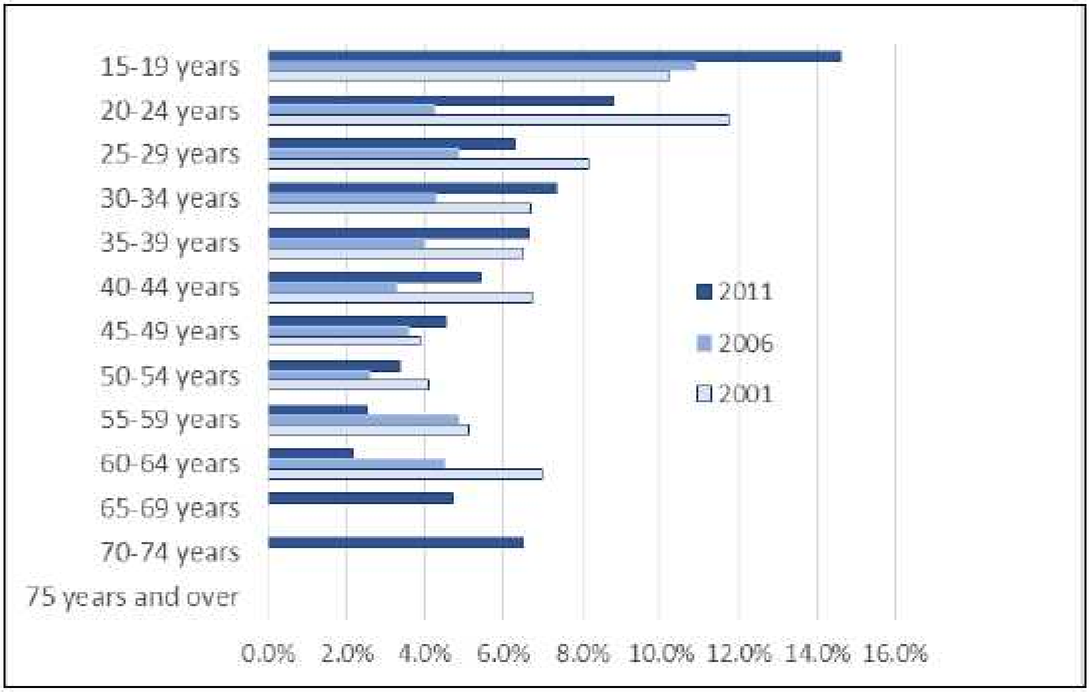

Figure 20-6: Unemployment rates – Katherine LGA
Source: ABS data

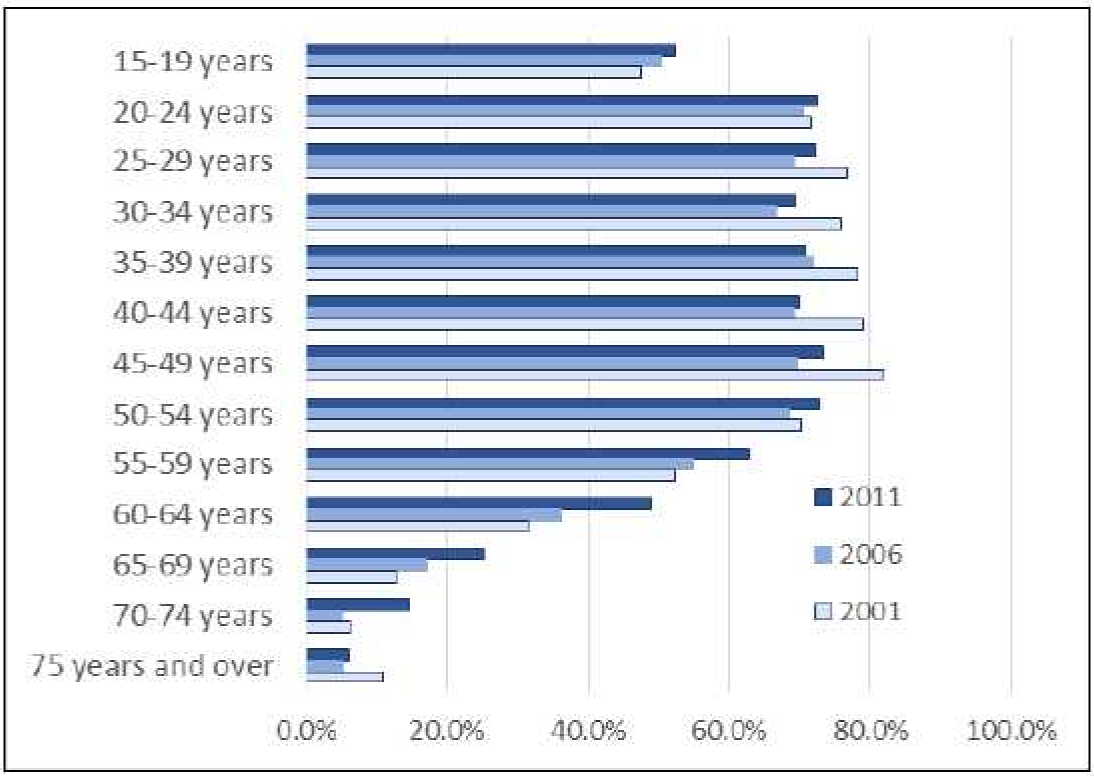

Figure 20-7: Labour force participation rate, Katherine LGA (ABS data)

Median weekly household income reported at the last census in the Katherine LGA was approximately AUD1,424. Median weekly rental and mortgage repayments at the 2011 census were AUD200 and AUD433, respectively. Current median rentals range from around AUD320/ week for a unit to AUD450/ week for a house (https://www.realestate.com.au/neighbourhoods/katherine-0850-nt, accessed 14 Jan 2016). Slightly over 50% of the Katherine population lives in rented accommodation.

Approximately 3,860 private dwellings exist in the Katherine municipality, of which about 8.5% (329 dwellings) were unoccupied at the time of the 2011 census. Housing types include single dwellings (~58%); semi-detached structures, townhouses, units and apartments (~17%); and a range of other accommodation, such as caravans, tents and improvised accommodation. Northern Territory government budget forecasts predict continuing strong demand for housing in Katherine, with relatively low vacancy rates (NT Government budget, 2013-2014). The Town of Katherine Land Use Plan (2014), anticipates the need for an additional 81ha of urban residential land and 180 ha of rural lifestyle lots (rural, rural living and rural residential) to accommodate residential development to beyond 2026. In the main, the areas being considered for additional residential development lie to the south and northwest of the Katherine urban centre and are unlikely to encroach on the Maud Creek project area, however evolving land use patterns may need to be taken into account as part of transport planning for the project.

The Katherine community comprises a number of different groups, including non-resident tourists and seasonal workers, part-time residents (FIFO workers), employees of the RAAF Base Tindal and their families, and full time residents. Stakeholder engagement for the Maud Creek project will need to take account of these diverse stakeholders, as well as the needs and expectations of Aboriginal traditional owners and their representative bodies, along with the requirements of government stakeholders at local, regional and territory level.

20.1.2 Surface Water

The Maud Creek Project area is traversed by Gold Creek, an ephemeral tributary of Maud Creek, which flows into the Katherine River upstream of the water supply extraction point for the Katherine Township. The distance between the proposed mine operations area and the confluence with the Katherine River is approximately 10.5 km. The Katherine municipality sources its water from a combination of groundwater bore and the Katherine River. The town water supply use of water from the Katherine River is sometimes constrained by the high turbidity in surface watercourses during the wet season. Neither Maud Creek nor Gold Creek contribute significant (or possibly, any) flow to the Katherine River during the dry season, although remnant pools persist along the watercourses throughout the dry season. Hydrological modelling conducted in connection with Terra Gold' s Maud Creek proposal (discussed in Section 4.5) concluded that the channels of both Gold Creek and Maud Creek would be likely to experience over bank flows on several occasions in every wet season. Modelling conducted by SRK is consistent with the earlier assessment (Figure 20-8). Surface water quality in the project area has been characterised to a limited extent. Dry season sampling conducted by previous tenement holders between 1968 and 1998 recorded generally good quality water, of near-neutral pH, relatively low salinity and low concentrations of most trace metals (URS, 2008). The EIS prepared in connection with the Terra Gold Maud Creek proposal suggested that surface waters in the project area may occasionally show elevated concentrations of copper or lead, presumably as a result of mineralization in the catchment, however additional monitoring would be required to demonstrate this convincingly.

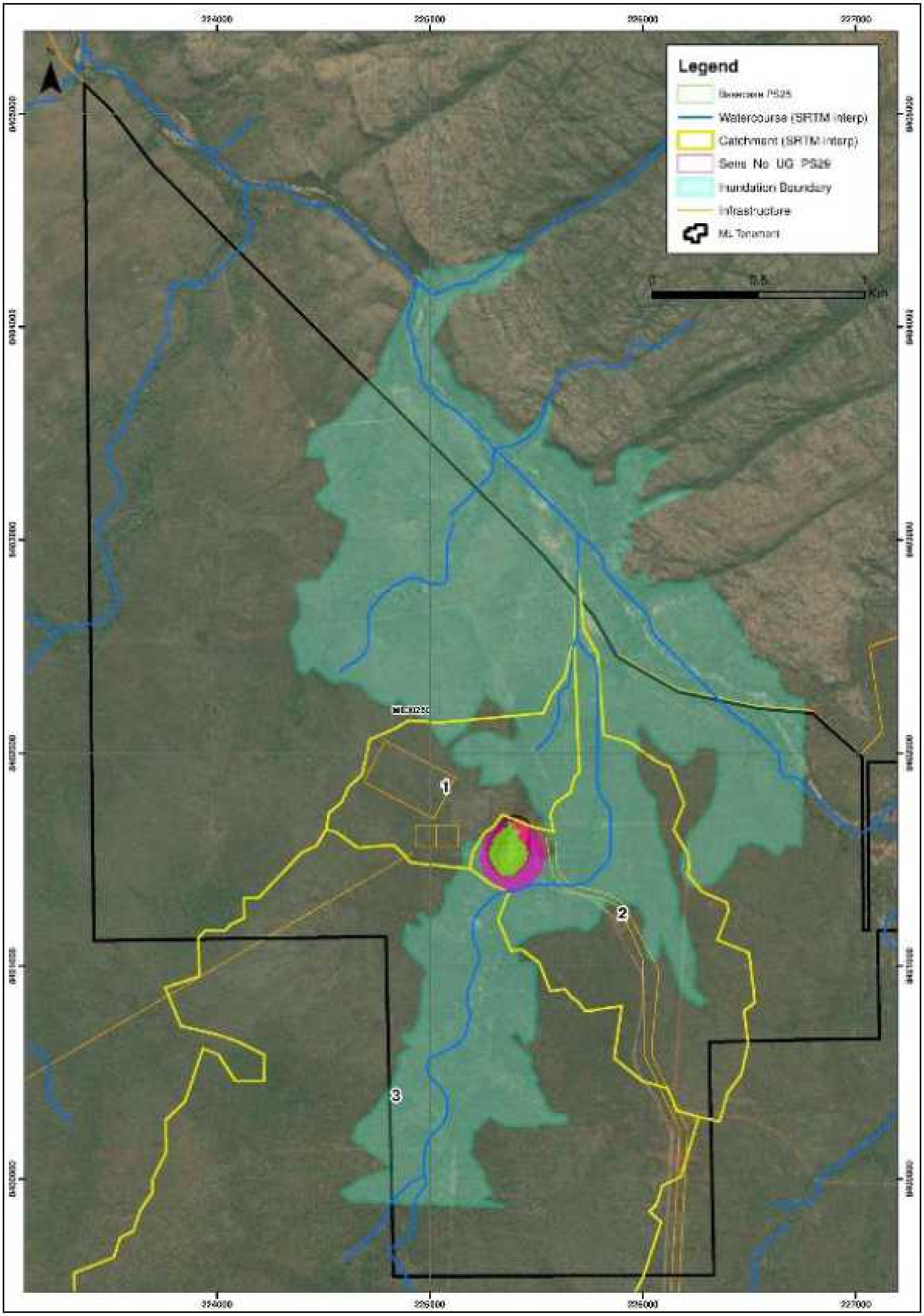

Figure 20-8: Estimated flood extents in absence of engineering controls

20.1.3 Groundwater

Groundwater beneath the Maud Creek site occurs primarily within fractured rock (Tollis formation) aquifers and unconsolidated alluvial sediments. Groundwater recharge is expected to occur from infiltration of rainfall, especially in areas of outcropping quartz / quartz breccia. Some recharge would also occur seasonally through the bases of watercourses. The proposed mine operations area lies to the north of the Tindall limestone formation, upon which the Town of Katherine, and a range of local enterprises, rely for water supply (Figure 20-9).

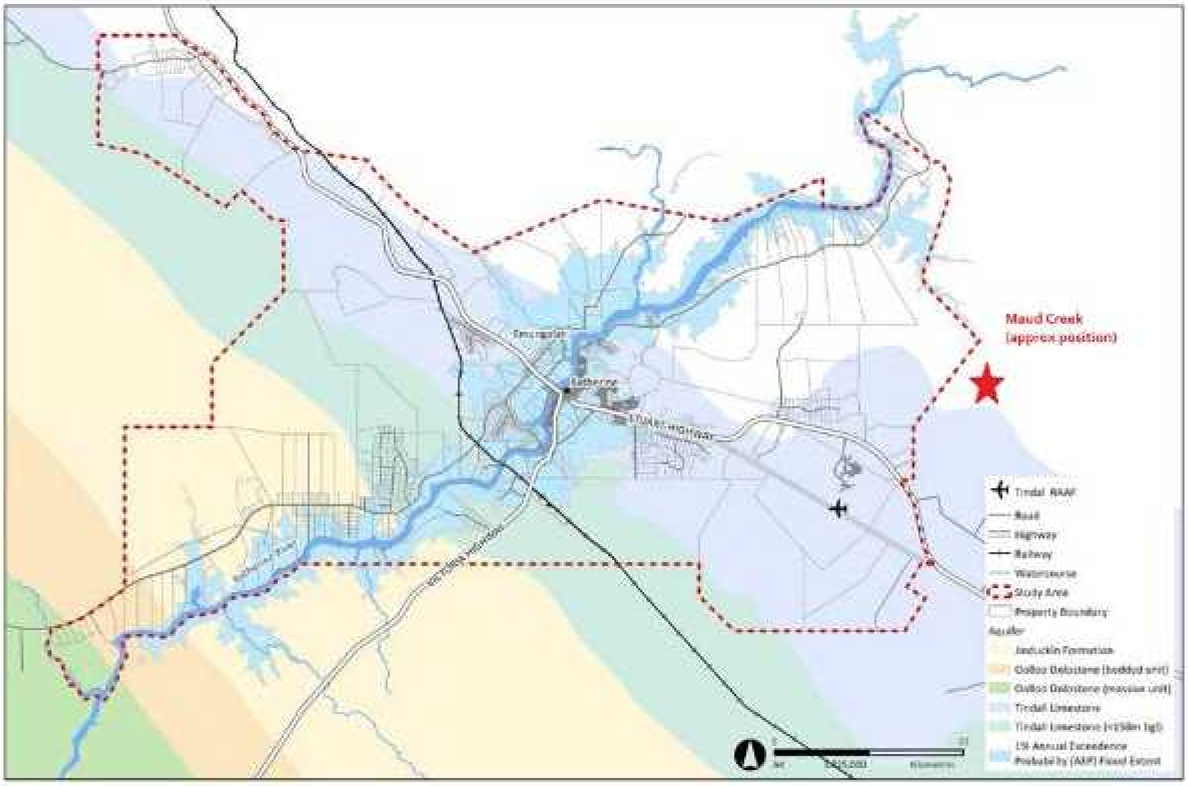

Figure 20-9: Groundwater aquifers near Katherine

Source: Katherine land use plan, 2014

The groundwater table in the Maud Creek area generally lies at shallow depth (between 1 and 6m below ground surface, corresponding to an average elevation of approximately 141m AHD). The seasonal variation in the depth to water is expect to be in the range of 2 to 4m. The general direction of groundwater flow is to the northeast, away from the Tindal limestone aquifer.

Baseline monitoring conducted in connection with Terra Gold' s EIS (URS, 2008) reported the dry season groundwater level at the confluence of Gold and Maud Creeks to be approximately 6.9 m below ground level (approximately 4 m below the creek bed). Limited if any contribution of groundwater to local stream flow is likely during the dry season, as the groundwater levels will generally lie below the beds of the watercourses.

Groundwater quality is typically fresh, with total dissolved solids concentrations below 600 mg/L. Groundwater chemistry is dominated by calcium and magnesium bicarbonates and the groundwater pH is slightly alkaline, with an average pH around pH8. Information in Terra Gold' s Maud Creek EIS suggested that the local groundwater may have naturally elevated concentrations of arsenic and/or selenium, however this conclusion appears to be based on limited water quality testing and would need to be confirmed through further groundwater monitoring.

20.1.4 Native Flora and Vegetation

As part of the EIS, Terra Gold commissioned field surveys of flora and vegetation for its Maud Creek proposal. The botanical investigations, which were carried out in March and April 2007, identified 153 plant species of which 19 were introduced weeds. None of the 153 plant species recorded during the 2007 were listed under the Commonwealth Environment Protection and Biodiversity Conservation Act 1999 (EPBC Act) as vulnerable, threatened, rare or endangered. One species, Tephrosia humifusa, recorded at 5 sites during the field surveys, is listed as Near Threatened (NT) under the Territory Parks and Wildlife Conservation Act 2006. The 2007 survey reported noted that Tephrosia humifusa appeared to be common locally, occurring at over one quarter of the floristic study sites and in a range of different habitat types. At the time of the Terra Gold EIS, Tephrosia humifusa was known to occur at 5 different locations in the Northern Territory. The Maud Creek survey records represented a range extension of the species. Current government records shown 19 occurrences of Tephrosia humifusa in the Northern Territory (not including at Maud Creek, refer Figure 20-10). The plant remains on the Northern Territory endangered species list (2012).

Figure 20-10: Distribution of *Tephrosia humifusa*

Source: Atlas of Living Australia, accessed 15/01/2016

Northern Territory government databases show a number of protected flora species in area to the north of the Maud Creek project area, but none within the mine tenement ((data from http://nrmaps.nt.gov.au/) (Figure 20-11).

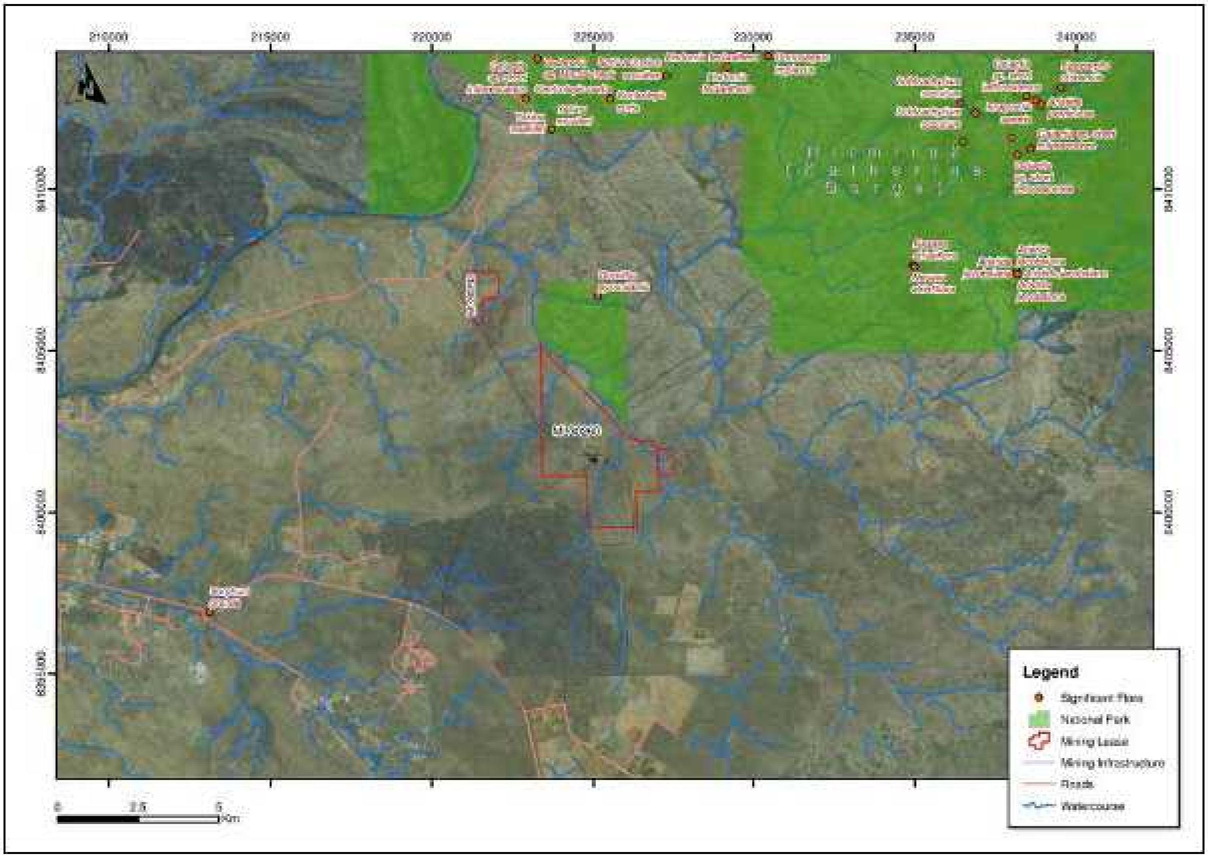

Figure 20-11: Conservation significant flora Source: http://nrmaps.nt.gov.au/)

Ten distinct vegetation communities were mapped in the Maud Creek Project area during the 2007 surveys (Figure 20-12). The vegetation communities included open forest, woodland and open woodland assemblages. None of the vegetation types was considered of special conservation significance: all occur widely outside the project area and none is thought to support specially protected flora or fauna species. No vegetation communities protected under the EPBC Act were recorded during the 2007 surveys. There is currently no mechanism for listing Threatened Ecological Communities under NT legislation. However, one threatened ecological community that occurs in the general project locality is listed as Endangered under the Commonwealth EPBC Act. This is the Arnhem Plateau Sandstone Scrubland Complex, which has had protected status since 2011. The Arnhem Plateau Sandstone Scrubland Complex is known to occur in parts of the Nitmiluk (Katherine Gorge) National Park. It not likely to occur within the Maud Creek mine tenement. This would need to be confirmed as part of future project assessment and permitting.

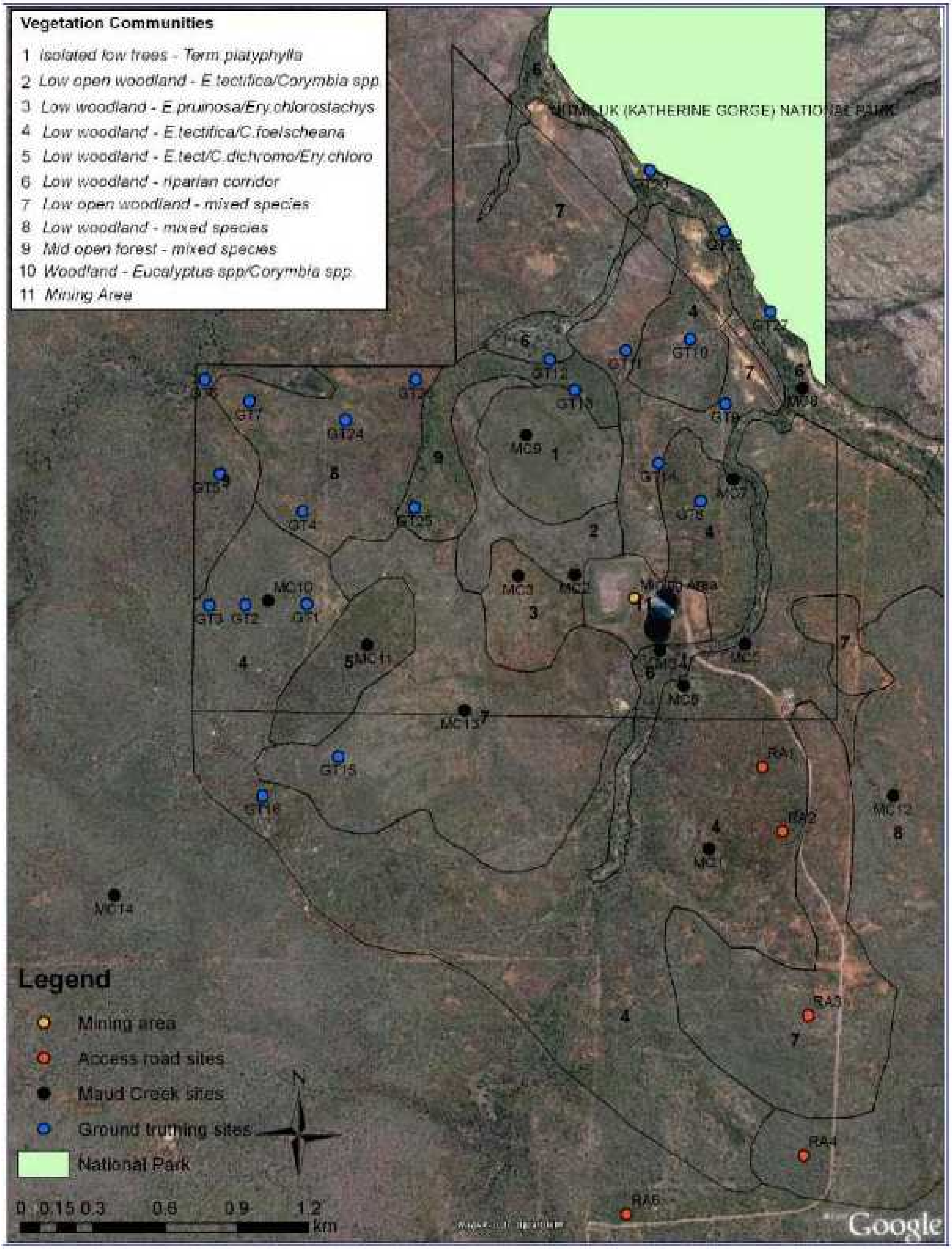

Figure 20-12: Vegetation communities Source: Crawford & Metcalf, 2007

20.1.5 Native Fauna and Habitats

Terrestrial fauna habitats

The most recent field fauna surveys of the Maud Creek project area were conducted in April and May 2007. Previous surveys had been carried out in 1994, 1996 and 1997 (EMS, 2007). The 2007 surveys recorded a range of habitat types, including open woodland, grassland and riparian habitats. Distinctive and geographically limited habitat types underlain by limestone karst or limestone outcrops were noted to the west of the mine tenement, in an area that had been proposed for an access road. No limestone outcrop was recorded within the mine tenement itself. Overall, the fauna habitats within the proposed mine operations areas were not characterised by significant biodiversity. Many areas had been disturbed by past mining activities, grazing pressure, weed incursion and disturbance from swamp buffalo, cattle, pigs, feral donkey and fires. Riparian habitats, which are not locally extensive, have been degraded by large numbers of swamp buffalo. Habitats within the mine tenement are unlikely to support significant biodiversity. None of the habitats present in the project are likely to provide significant habitat for water birds or shore birds. The limestone outcrop habitats to the west and south of the mine operations areas are less heavily impacted and may act as locally significant refuge and areas that support conservation significant wildlife, including bats and a variety of invertebrate species such as land snails.

Terrestrial vertebrate fauna

A total of 144 native and six introduced terrestrial vertebrate species were recorded during the 2007 surveys. These included 30 mammals, 91 birds, 18 reptiles, and 11 amphibian species. A number of varanid (monitor lizard) species observed during previous surveys of the area were absent during the 2007 surveys. This may be the result of predation and displacement by cane toads, which were introduced to the area in about 2001. Cane toads were the most common amphibian observed during the surveys.

The majority of fauna recorded or expected to occur in the Maud Creek Project area are widespread in northern Australia. Two bird species of conservation significance – the red goshawk (*Erythrotriorchis radiatus*) and the Australian bustard (*Ardeotis australis*) have been observed within the mine tenement. Both of these are relatively widespread within the NT. Six species that are considered Near Threatened under the according to the Territory Parks and Wildlife Conservation Act 2006 were also recorded in or very near to the proposed mine operations area ((data **from http://nrmaps.nt.gov.au/)**

Figure 20-13). They included: the bush stone-curlew (*Burhinus grallarius*), hooded parrot (*Psephotus dissimilis*), grey falcon (*Falco hypolelucos*), northern nailtail wallaby (*Onychogalea unguifera*), Arnhem sheathtail bat (*Taphozous kapalgensis*), orange leafnosed bat (*Taphozous georgianus*) and western chestnut mouse (*Pseudomys nanus*). No significant migratory bird species were recorded on or near the mining tenement. (The EPBC-listed rainbow bee-eater (*Merops ornatus*) was recorded, but this species is very common and is unlikely to be significantly impacted.)

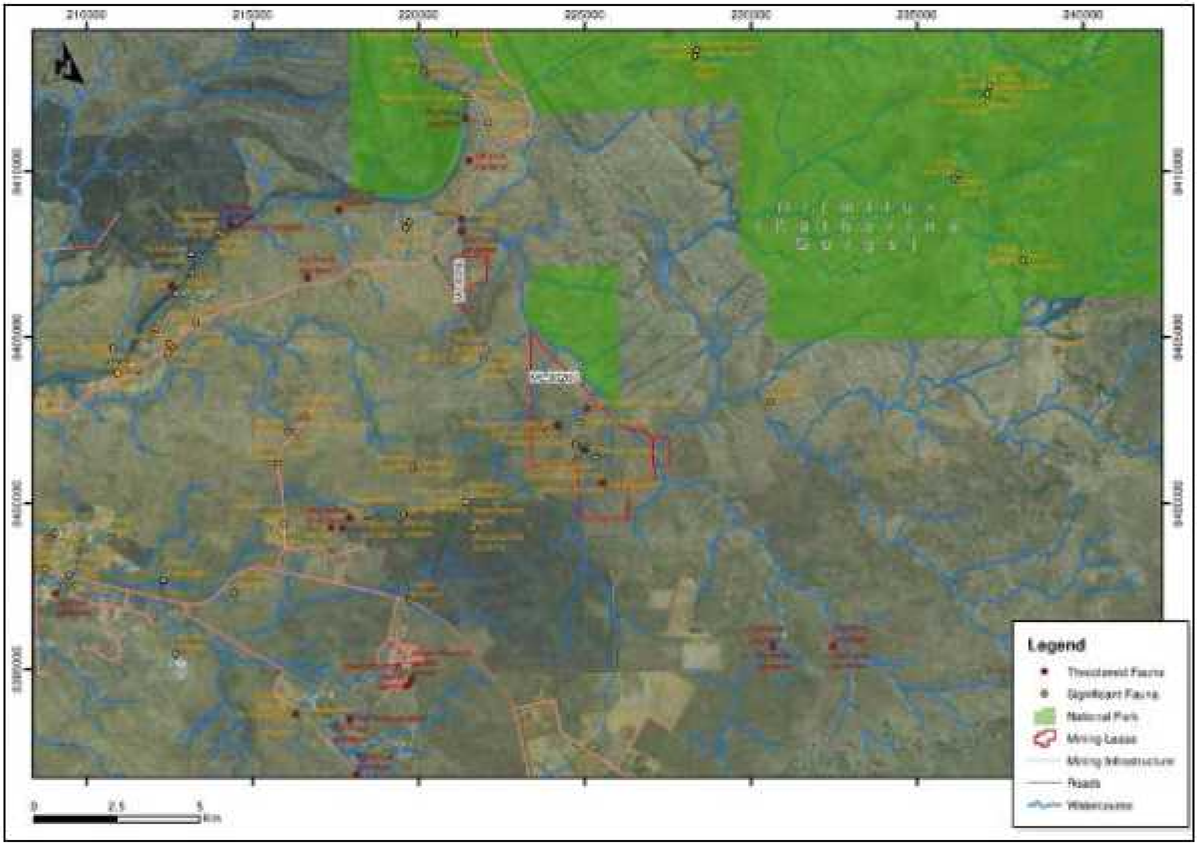

Figure 20-13: Conservation significant fauna

Source: http://nrmaps.nt.gov.au/)

Terrestrial invertebrate fauna

Land snails were collected and identified at all the limestone outcrop areas surveyed to the west of the mine tenement (along the access road alignment previously proposed by Terra Gold). Some of the snails observed in the limestone outcrop habitats (*Xanthomelon* sp) were also collected from open forest habitats in the project area. This species is thought to be relatively widespread, but occurs at low densities within its range (EMS, 2007). The most common snail species collected was *Torresitrachia weaberana,* a moderately commonly species known to occur in areas between Katherine and Kununurra. One of the snail species collected was an undescribed genus and undescribed species that is known to be limited in range to limestone karst in the Tindal - Cutta Cutta region. This species was be considered to be of regional conservation significance. Recent technical studies on snails in the Katherine region have proposed that endemic land snails can be used as bio-indicators for the health of some ecological assemblages and may themselves be under threat of extinction (Braby *et al*, 2011, Willan *et al*, 2009).

Aquatic fauna and habitats

Fifteen species of freshwater fishes were recorded in Gold Creek and Maud Creek during the 2007 field surveys. The western rainbowfish (*Melanotaenia australis*) was the most abundant species in samples and was the only species present at all sites. Other common species included the banded grunter (*Amniataba percoides*), spangled grunter (*Leiopotherapon unicolor*) and northern trout gudgeon (*Mogurnda mogurnda*) (URS, 2008).

No listed threatened aquatic species were recorded in recent or previous field surveys in the project area. Although the EPBC Act lists the freshwater sawfish (*Pristis microdon*) as a vulnerable species that could potentially occur in the vicinity of the project area, no suitable habitat for this species occurs in proximity to the proposed mine operations it. Neither is the species known to occur in the Katherine River.

None of the freshwater fishes recorded during past surveys of the Maud Creek project area are listed as threatened in the NT or under Commonwealth legislation. The species occurring Maud Creek are all common and widespread forms that are well adapted to the variable instream conditions that characterise the system Aquatic habitat quality along Gold Creek was described as ‘significantly impaired’ by trampling and other disturbance by swamp buffalo, donkeys, feral pigs and cattle. Neither Maud Creek nor Gold Creek are considered likely to support high levels of biodiversity in the project area, although the riparian habitats may afford a level of shelter and temporary feeding and growing habitat. The watercourse may provide a migration path for aquatic species during the wet season (URS, 2008).

20.1.6 Air Quality, Noise & Vibration

The proposed mine operations area is located at least 5 km from the nearest ‘sensitive receptor’ (residential premises, for example). Providing conventional environmental controls on dust and noise emissions are implemented during construction and operations, it is unlikely that significant impacts on environmental amenity will arise.

There is some risk that offsite traffic associated with the project will give rise to public concern. Arrangements for controlling impacts of product haulage, workforce traffic and vehicle movements for transport of fuel and reagents will need to be described as part of the project’ s environmental impact assessment.

20.1.7 Conservation Areas

The closest conservation area to ML30260 is the Nitmiluk (Katherine Gorge) National Park, owned by the Jawoyn Aboriginal Land Trust and jointly managed with the Parks and Wildlife Commission (Figure 20-14). The park is notable for its scenery, cultural values and ecological integrity. The proposed mining operations are unlikely to give rise to direct impacts on Nitmiluk National Park, although there is some potential for indirect impacts on the park, particularly if the Maud Creek Project includes the upgrade and use of the general service easement running north from the mine to Gorge Road. Increased use of the northern access route has the potential to increase the risk of spreading weeds and/or pathogens and may exacerbate the adverse environmental impacts caused by animal pests. Improved road access could also increase the risk of bushfires, by making the area generally more available to the public.

It is unlikely that the mining operations would intrude on parts of the park that are routinely accessed by visitors, although some consideration of light spill and noise (blasting, offsite traffic) may be required as part of a future environmental impact assessment.

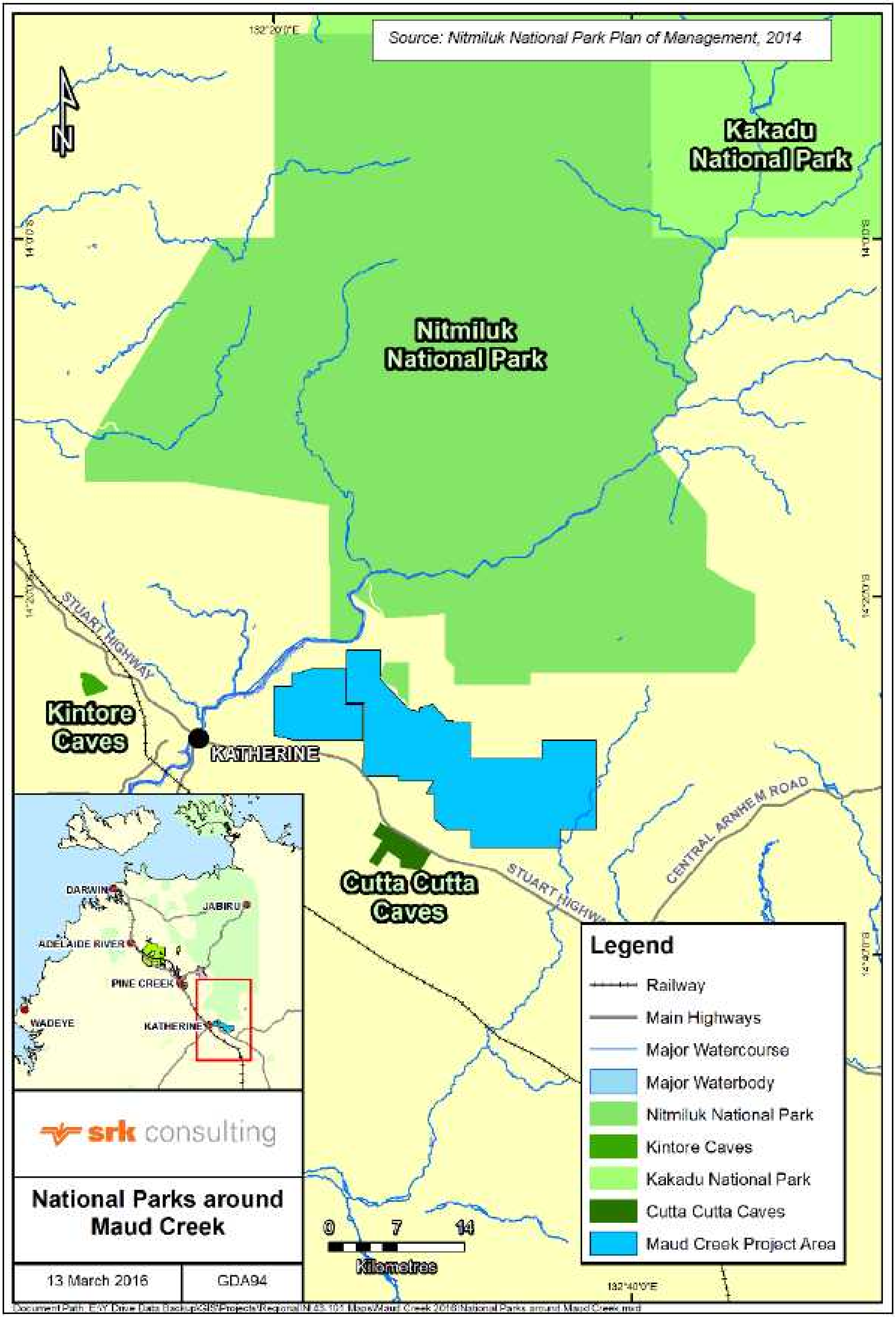

Figure 20-14: Locations of parks and reserves near Maud Creek

20.1.8 Heritage Values

Heritage surveys relating to both Aboriginal and non-Aboriginal values of the Maud Creek mine site were conducted as part of baseline studies for the Maud Creek EIS prepared by Terra Gold (URS, 2008, refer Figure 20-15).

The studies identified sixteen Aboriginal archaeological sites, including 14 stone artefact scatters and three stone quarries (one the quarries was associated with an artefact scatter). Each site was rated as having high, moderate or low significance, based on rarity, intactness and age of the site. In all, over 240 stone artefacts were identified. Sixteen of the artefact scatters were considered to have high significance, three were classified as having moderate significance and five were described as having moderate to low significance. Two of the three stone quarries were described as highly significant.

A number of additional sites were identified on the Maud Creek tenements subsequent to the completion of Terra Gold' s EIS. The current Newmarket Gold Cultural Heritage Environmental Management Plan (2015) identifies 18 Aboriginal sacred sites. Disturbance of any Aboriginal sacred site would require formal authorisation (refer Section20.2.8) ..

Two non-Aboriginal heritage sites were identified during baseline surveys for Terra Gold' s Maud Creek project. One site comprised a series of historic alluvial gold mine diggings at the eastern side of the project area. The site was described as having significant heritage value. The second non-Aboriginal heritage site was a feature from an historic settlement to the northeast of the project area. The site consists of a raised stone hearth and a foundation made from cement, gravel and earth and edged with cobbles of rock. A variety of other artefacts were identified (shards of stoneware and earthenware, preserved meat tins, tobacco tins and matchboxes). These were generally rated as having a moderate to low significance.

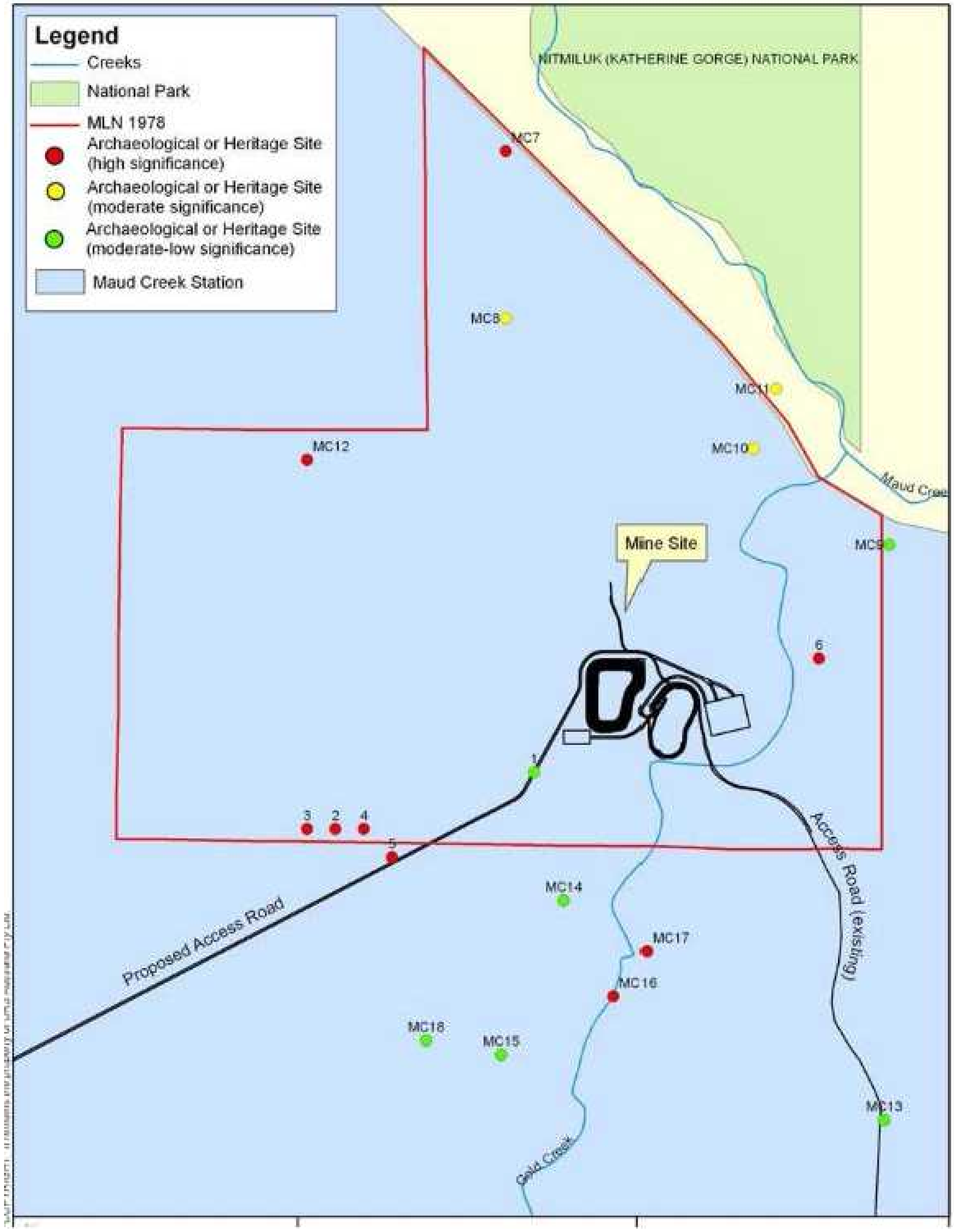

Figure 20-15: Aboriginal and non-Aboriginal heritage sites
Source: Figure 11.1 from URS, 2008

20.1.9 Mine Closure and Revegetation

The EIS prepared for Terra Gold' s Maud Creek Project included a preliminary discussion of mine rehabilitation objectives and outcomes. The approach proposed by Terra Gold was to rehabilitate disturbed land to allow future use primarily for pastoral purposes, with some areas (such as waste rock dumps) potentially allocated for conservation purposes. It was suggested that water in pit voids might be used for watering livestock. The closure strategy also proposed to use pit voids for disposal of reactive materials remaining at the ROM. No closure provision was made for rehabilitation of tailings or waste rock storage facilities, as the project did not include these elements.

A significant review of mine rehabilitation and closure will be required as part of project assessment and permitting, both to reflect changing community expectations and to align with evolving government regulations and policy. Long-term management of mine wastes and of water in pit voids is likely to be a central issue for mine closure design. Newmarket Gold should expect an increased focus on verifiable rehabilitation performance targets.

20.1.10Potential Impacts

Although a range of environmental impacts is possible as a result of implementing the Maud Creek Project, the aspects most likely to attract the attention of stakeholders are:

- Protection of surface water and groundwater quality, especially from contaminants contained in tailings and other mineral wastes;
- Avoidance of culturally significant areas;
- Potential for impacts on public safety and/or amenity from mine traffic in or near the town of Katherine
- Control of indirect impacts (weeds, fire, feral animals) on the environmental values of Nitmiluk National Park
- Newmarket Gold' s ability to deliver acceptable mine rehabilitation outcomes

A summary of project aspects and impacts is presented in Table 20-1.

A more detailed analysis and formal risk assessment of these aspects and impacts will be required as part of project assessment and permitting.

Table 20-1: Preliminary aspects and impacts analysis

	Social / economic	Public health & safety	Aboriginal heritage	Surface water	Ground water	Soil quality / land capability	Flora / vegetation	Fauna & habitats	Air quality	Noise & vibration	**Ke**
Land access & clearing for mine development			**X**	**X**			**X**	**X**	**X**	**X**	Need mini footp overa imple of wo patho hygie proce limit impa ripar syste avoid sacre opera contr mini gene erosi noise
Land access & clearing for access road(s)			**X**	**X**			**X**	**X**	**X**	**X**	Need confi avoid impa mini to sh ende (eg la and h conse value vege asser Most a sig issue acces not a
Mine dewatering	**X**			**X**	**X**						Stron poter publi abou conta (disc wate abstr wate
Mine operations: haulage &								**X**	**X**	**X**	Unli gene signi impa

traffic within mine site											conc routi opera pract mini footp contr dust.
Mine operations: offsite haulage & traffic	**X**	**X**							**X**	**X**	Has p to ge signi conc relati traffi (dust publi amen impa publi infra
Blasting, excavation, heavy equipment operation									**X**	**X**	Unlil gene signi impa routi opera pract mini footp contr dust.
Handling and stockpiling of waste rock				**X**	**X**	**X**			**X**		Key likely to i) mana react gene erodi mate ii) ab rehal land
Ore processing				**X**	**X**				**X**	**X**	MMI need demo appro syste contr noise proce efflu
Tailings storage	**X**			**X**	**X**						Key likely to i) mana react

											gene salin meta or er mate abilit rehal land; of da (espe with surfa impa durin mine opera
Establishment and operation of mine accommodation	**X**										May conc hous avail cost; impa relati socia integ incre traffi poter socia
Waste generation (non- process wastes)				**X**	**X**						Unli a ma conc prov conv envir mana contr place hand dispo sewa dome and i waste

20.2 Regulatory Approvals

20.2.1 Mineral Titles Act

With the exception of certain prescribed substances (such as uranium), all minerals located in the Northern Territory (except for certain prescribed substances including uranium) are the property of the Northern Territory Crown. The Mineral Titles Act took effect in November 2011, replacing the Mining Act. The Mineral Titles Act provides a framework for granting and transferring mineral titles to authorise exploration for and extraction of minerals. It also establishes a basis for authorising ancillary activities related to mining.

The Mineral Titles Act recognises seven categories of mineral title, of which the most relevant to the Maud Creek Project is a mineral lease (ML). Newmarket Gold was granted a mineral lease (ML30260) over the project area on 14 April 2014. The lease over the 106 ha tenement is valid until 13 April 2024 and can be extended, subject to certain conditions (including expenditure requirements and compliance with tenement conditions).

Grant of a mineral lease gives the tenement holder exclusive rights for the development of mineral projects on land within the title boundaries. Subject to the tenement holder obtaining an authorisation under the Mining Management Act, the grant of a mineral ease confers rights to conduct mining and processing of minerals, as well as a range of mining related activities, including:

- exploration for minerals;
- treatment of tailings
- storage of waste

Additionally, the Mineral Titles Act provides the holders of mineral titles strong rights of access and important entitlements related to the taking and use of water. For example, a mineral title holder may apply for an access authority to enter land which is outside the title area for the purpose of constructing infrastructure or for the taking of water.

The Mineral Titles Act also provides a basis for assigning liability for mine rehabilitation. An entity taking ownership of a tenement on which disturbance exists (as is the case at Maud Creek) assumes the risks and responsibility for that tenement, irrespective of whether it causes any additional mining disturbance to the land.

20.2.2 Mining Management Act

Before any mining activity can occur on a granted mining title the intending operator of the mining activity must apply for and be granted an authorisation under the Mining Management Act. The Mining Management Act (MMA) effectively constrains rights conferred by the Minerals Title Act (MTA) by prohibiting mining activities until the intending operator (which may or may not be the tenement holder) has:

- been granted an authorisation (supported by an approved mining management plan (MMP)), and
- lodged a security to cover the full costs of rectifying any environmental harm arising from mining activities and for final rehabilitation of the affected area (including rehabilitation of any legacy disturbance).

Authorisations granted under the Mining Management Act typically cover all mining (and related) activities required for project implementation. In some circumstances, separate authorisations may be granted for activities (operation of an explosives storage facility, accommodation village, power station) which the mine operator does not itself wish to operate and manage.

Mining management plans (MMP) are binding legal instruments through which pollution and environmental harm are controlled on mine tenements. Operators of mining projects are exempt from some licensing requirements that would normally apply to comparable activities on non-mining land. The MMP instead serves regulate resource usage (taking of water) and other environmental aspects of mining (and related activities) within the bounds of the granted tenement. It is an offence to contravene the environmental obligations arising from an approved MMP, if the activities result in environmental nuisance, harm or pollution. More generally, the Act prohibits the release of wastes or contaminants unless the release is done in accordance with an approved MMP.

20.2.3 Waste Management and Pollution Control Act 2009

The Waste Management and Pollution Control Act is administered by the NT EPA. The Act regulates the collection, transport, storage, treatment and disposal of listed wastes. Wastes – including both process and non-process wastes – arising from mining activities are generally exempt from licensing requirements under this Act and are instead regulated through the approved mining management plan.

20.2.4 Water Act 1992

Mining operations in the Northern Territory are not required to seek licences for abstraction and use of water under the Northern Territory Water Act, providing their use, storage and management of water is done in accordance with an approved mining management plan. Mine tenement holders are entitled to construct bores, and to take or divert water (not including water in dam or wells belonging to others) and to use the water for mining and mineral processing.

Pollution provisions of the Water Act generally do not apply to mining operations, except in the event that contamination or pollution moves beyond the tenement boundary (either through an intentional or accidental discharge). Mine operators are exempt from discharge licensing requirements that would normally apply under the Water Act, to the extent that:

- the contaminant or waste results from carrying out of a mining activity;
- the waste or contaminant discharge is confined within the land on which the mining activity is carried out; and
- the discharge is done in accordance with an approved mining management plan.

In circumstances where a discharge of a waste or contaminant (for example, water from mine dewatering) is likely to move beyond the boundaries of the mine tenement, a discharge licence would be required. As an example, Newmarket Gold holds a waste discharge licence (WDL 138-02) to authorise release of wastewater from its Union Reefs site to Wellington Creek and the McKinlay River. Newmarket Gold has implemented comprehensive environmental management systems for several of its NT operations: similar arrangements would be appropriate at Maud Creek.

A waste discharge licence may be required for activities that could result in seepage that could cause pollution of an aquifer. The implications of this for storage of tailings and waste rock and for the development of permanent pit voids would need to be discussed with NT regulators.

20.2.5 Aboriginal Land Rights (Northern Territory) Act 1976 (Cwlth)

Contemporary use of the term 'traditional owner' largely derives from the Aboriginal Land Rights (Northern Territory) Act 1976 (ALRA). The ALRA established ways for Aboriginal people to claim land in the territory on the basis that they were the traditional Aboriginal owners of the land. Further, the ALRA gave traditional Aboriginal land owners the right to control access to their land, including the right to withhold consent for exploration activities on their land. This Act has limited, if any, relevance to the Maud Creek Project, as mining tenure has already been granted and the project lies on freehold land that is not subject to the control of any Aboriginal Land Trust.

20.2.6 Native Title Act 1993 (Cwlth)

The Native Title Act 1993 establishes a legal framework for the recognition and protection of native title. It also establishes an administrative system for:

Determining claims to native title;

Validating past acts which might otherwise be invalidated because of the existence of native title; and

Developing processes and standards for future dealings which could affect native title.

The Maud Creek Project is located on freehold land, and therefore is not exposed to native title claims. There are no registered or determined native title claims over the project area.

20.2.7 Heritage Act 2011

The *Heritage Act 2011* declares all Aboriginal archaeological places and objects heritage places. Other historically or culturally significant places (for example, buildings and heritage objects) are also protected under the *Heritage Act* if they are listed on the NT Heritage Register. No non-Aboriginal heritage sites formally registered on the Australian Heritage database are known to occur in the project area. No formal authorisation would be required in relation to disturbance of the non-Aboriginal heritage sites and Terra Gold made no commitments to the protection or conservation of these sites.

The Act makes provision for approval of works to salvage Aboriginal archaeological sites, subject to the consent of the appropriate Traditional Owner or Site Custodian. It is normally a requirement that appropriate studies be conducted to assess the character, extent and value of a site before salvage. A person proposing to disturb a heritage site would be expected to commit to a program to prevent or minimise damage and to make provision for suitable curation of any artefacts. For Aboriginal sites, suitable curation typically involves return of the object(s) to the Traditional Owners or protection if artefacts are outside of the operational footprint.

20.2.8 Northern Territory Aboriginal Sacred Sites Act 1989

The *Aboriginal Sacred Sites Act* provides the legal basis for protection of all Aboriginal sacred sites, irrespective of whether the site is formally registered. Carrying out work – or even accessing – a sacred site requires a consent in the form of a certificate issued by the administering agency, the Aboriginal Areas Protection Authority (AAPA).

A certificate (reference number D89/199:90/804 (Doc: 68552) C2009/266 (supersedes C2007/072)) was issued by the AAPA to Crocodile Gold on 8 October 2009. The certificate authorises access to / disturbance of nominated Aboriginal sites within the Maud Creek mine site (ML30260 – formerly MLN1978), as well as some other sites on other Newmarket Gold tenements named on the certificate. The consent is subject to a range of implementation conditions, including a condition excluding works or access to designated sites. No works of any kind are permitted at sites SS5369-69 (restricted work area 1), 5369-32 (restricted work area 2) or 5369-27. Two of the exclusion areas (restricted work areas 1 and 3) lie ML30260, refer Figure 20-16.

A further implementation condition (Condition 3) says, *this certificate shall lapse and be null and void, if the works in question or the proposed use is not commenced within 24 months of this certificate*. Newmarket Gold has advised that some of the works approved under the AAPA certificate (specifically, exploration drilling) was carried out in 2011, prior to the expiry of the AAPA consent. Accordingly, the consent is still valid.

With regards to the NT Aboriginal Sacred Sites Act, Newmarket Gold have the requisite permit (AAPA Authority Certificate C2009/266 superseding C2007/072), provided the project footprint remains within the Subject land as defined in the Authority Certificate, and avoids the two exclusion areas, then no further permitting is required.

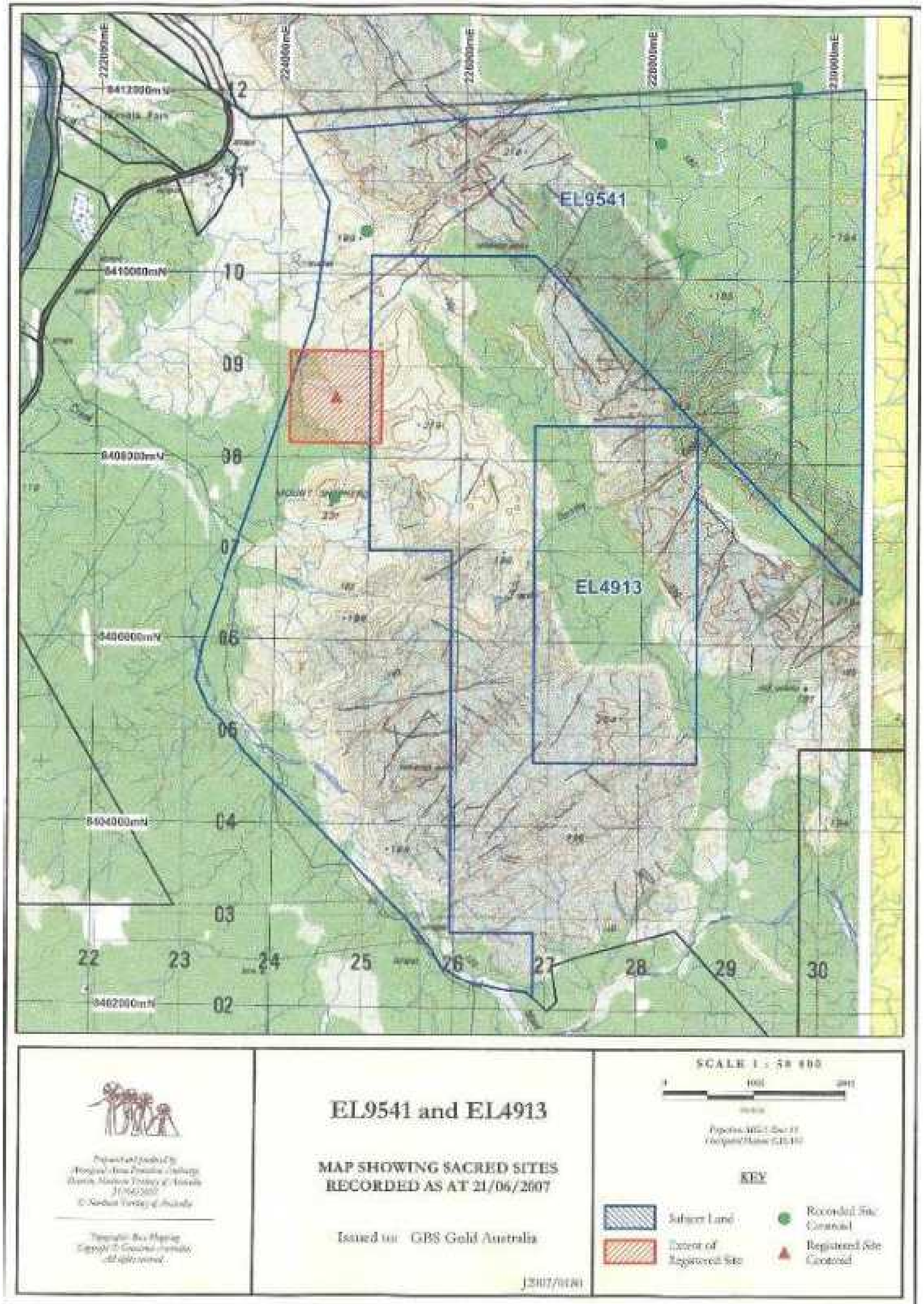

Figure 20-16: Restricted work areas

Source: Excerpt from AAPA certificate, June 2007

20.3 Waste Rock and Mineralization Geochemistry

20.3.1 Available Data

Drillhole Database

The current drillhole assay database (*Database_08_05_2015*) contains assay data for Au, As, Cu, Pb, Zn, Cr, and Ni. A smaller extended assay dataset was obtained from drilling in 2011, which includes Ca, Mg, Ag, Bi, Fe, Hg, K, Mn, Mo, Na, S, Sb, Sn, Ti and Zr.

Total sulphur can sometimes be used as an indicator of acid generating potential - the assumption being that all sulphur is present in the form of potentially reactive, acid generating sulphides.

The sulphur assay sampling density was low, with 2,140 samples collected from 21 drillholes (Figure 20-17) relative to the 2,129 drillholes included in the database.

The sulphur assay data was separated into waste-grade and ore-grade assuming a gold cut-off value of 0.1 mg/kg. Sulphur content in the waste rock (n=1,831) assays ranged from <0.05 -8.55%S (mean 0.16%S), while in the ore-grade assays sulphur ranged between <0.05 -10.7%S (mean 0.92%S) . In the waste rock lithologies the highest sulphur contents were present within the dolerite (8.55%S), sediments (SLST, 3.75%S), Cambrian Basalt (3.55%S), tuff (2.75%S) and veins (2%S).

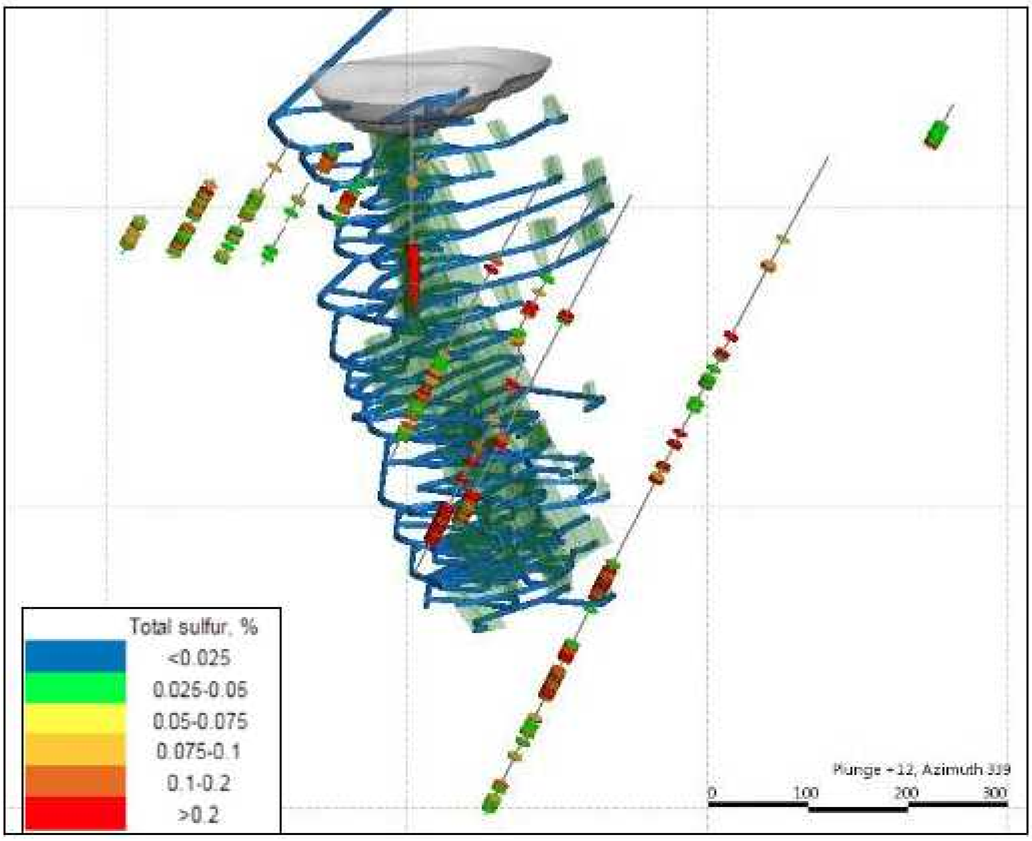

Figure 20-17: Drillhole database S assay data

Notes: The figure also shows the proposed open-cut pit (grey), underground workings (blue) and modelled deposit (green)

Detailed Geochemical Characterisation

Previous geochemical characterisation of waste rock (33 samples), mineralization (11 samples) from the weathered (or oxide) zone and the fresh (unweathered) zone, and soil (2 samples) from the Maud Creek deposit was undertaken in 1996 (Graeme Campbell & Associates, 1997). The assessment was prepared in support of an Environmental Impact Statement prepared by Kalmet Resources NL (Kalmet), to identify materials that may generate Acid Mine Drainage (AMD) and/ or impact water quality and revegetation.

The sampled materials included 33 waste rock samples, 11 mineralized samples and 2 soil samples. The waste rock and mineralization samples were collected from the weathered and fresh rock zones – a breakdown of the lithologies sampled is given in Table 20-2.

Table 20-2: Sampled materials (GCA, 1997)

Waste Rock/ Ore	Weathering Zone	Kalmet Lithology	SRK Lithology	No. of Samples
Waste Rock	Weathered Zone	Hanging wall Mafic Tuff	Tuff	10
		Footwall Sediments Undifferentiated	Sediments (SDST)	1
		Footwall Sandstone	Sediments (SDST)	3
	Fresh Zone	Hanging wall Mafic Tuff	Tuff	11
		Footwall Sediments Undifferentiated	Sediments (SDST)	4
		Footwall Sandstone	Sediments (SDST)	4
Mineralization	Weathered Zone	Stockwork Quartz - Tuff Hosted	Vein	1
		Stockwork Quartz - Sediment Hosted	Sediment (MSED)	1
		Massive Quartz - Tuff Hosted	Vein	1
	Fresh Zone	Stockwork Quartz - Graphitic Sediment Hosted	Vein (2), MSED (1)	3
		Stockwork Quartz - Tuff Hosted	Tuff	1
		Massive Quartz - Graphitic Sediment Hosted	Vein (1), Tuff (1)	2
		Massive Quartz - Tuff Hosted	Tuff (1), Vein (1)	2

The assessment included the following test work on all 44 samples:

pH and EC (paste/slurries; 1:2 solid : water)

Total S

Acid Neutralisation Capacity (ANC) [2]

Net Acid Producing Potential (NAPP)

The following additional testwork was carried out smaller sub-sets of samples:

Sulphate-sulphur (SO4-S) and sulphide-sulphur (14 samples)

Net Acid Generation (NAG) (15 samples)

Multi-element analyses, (12 samples) (digest not specified) including the following elements: Ag, Al, As, B, Ba, Bi, C, Ca, Cd, Co, Cr, Cu, F, Fe, Hg, K, Mg, Mn, Mo, Na, Ni, P, Pb, Sb, Se, Sn, Sr, Th, Tl, U, V, Zn)

Saturation-Extract (SE) tests (7 waste rock samples) to assess the soluble salts content (Na, K, Mg, Ca, Cl and SO4) and the solubility of arsenic and antimony.

20.3.2 Sample Representivity

The drill-hole assay samples with sulphur data are generally located in rock volumes that lie outside of proposed open pit and underground workings (Figure 20-17).

With respect to samples submitted for detailed characterisation, six of the uppermost samples occur within depths ranging from 2-3 m below ground level (mbgl) to 174-175 mbgl; and include 7 samples from the weathered (or oxide) zone and 19 from the fresh zone[3]. No samples have been collected below 175 mbgl.

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[2] Modified ANC test (unspecified number of tests on waste rock) using dilute sulphuric acid to measure readily available ANC (ANCC).

In conclusion, current datasets do not adequately represent the volumes to be mined as part of the currently proposed open pit and underground workings.

20.3.3 Geochemical Characteristics

Results generated from the detailed characterisation are summarised in Table 20-3. Note that, as discussed in the previous section, the samples studied do not give good representation of material that could be mined as part of currently proposed open pit and underground workings. However, the data obtained have value in that they can be used to infer some characteristics of materials located within the topmost 175m.

Most of the waste rock samples contained an excess of ANC and would be classed as NAF (Figure 20-17). Highest ANC was offered by the Hanging wall Mafic Tuff samples; however, note that only a portion (10-20%) of ANC was considered readily available for reaction. The balance of acid potential and neutralising capacity is closest in the case of the Footwall Sediments and Footwall Sandstone samples, including three samples that were classed as having uncertain acid generating potential and two samples that classed as PAF.

Multi-element analyses results were used to calculate Global Abundance Indices (GAI) to identify elements that are enriched relative to average crustal abundance concentrations. In both waste rock and mineralization, elements identified as being enriched included arsenic, antimony, chromium, nickel, tin, boron and silver.

Saturation-Extract (SE) tests (9 samples waste rock; 2 samples ore) gave alkaline extraction solutions (pH 8.6 -9.2) . Of the enriched elements listed above, only arsenic and antimony were included in the leach analysis suite - giving dissolved concentrations of arsenic up to 0.56 mg/L (one of the mineralization samples) and antimony up to 0.046 mg/L (one of the waste rock samples). The results suggest that both arsenic and antimony could be leachable from material to be mined at Maud Creek. [It is noted that the existing pit lake water was found to contain elevated arsenic concentrations (185-201 µg/L; relative to the ANZECC (2000) freshwater trigger value of 13 µg/L).]

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[3] More recent geological interpretation of the weathering zone extents would place 15 of the waste rock samples within the oxide (weathered) zone, 1 within the transitional zone and 17 in the fresh zone.

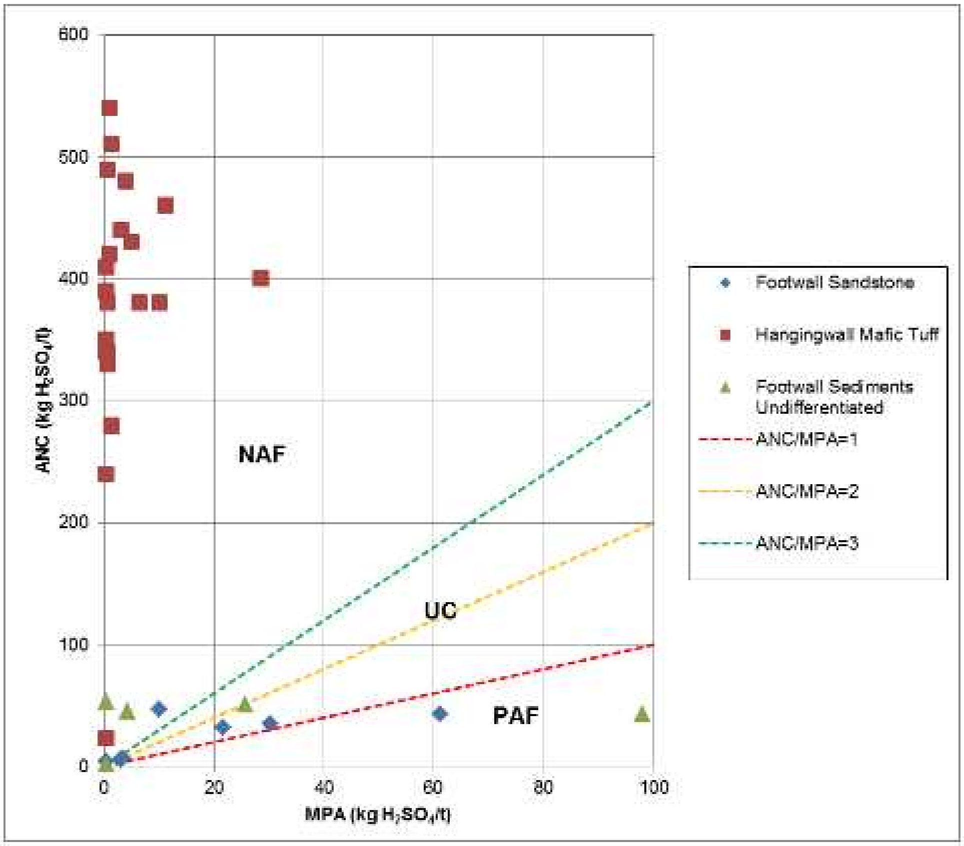

Figure 20-18: Geochemical classification of waste rock samples using NPR

Note: The plot includes lines to show where the NPR (ANC/MPA) values equal 1 and 3, indicating boundaries between PAF, UC and NAF regions on the plot. A line to show where the ANC/MPA value equals 2 is also shown, as values greater than 2 are considered to be unlikely to be problematic with respect to AMD (DITR, 2007).

Table 20-3: Summary of Available Geochemical Characterisation of the Deposit

Parameter	**Waste Rock**	**Mineralization**	**Comment**
Paste pH	7.6-9.4	6-9.1	
Paste EC, µS/cm	38-1800	34-490	
Total sulphur,%	<0.01- 2.1% <0.01- 0.11% (weathered zone) <0.01- 2.1% (fresh zone)	<0.01-3.2 <0.01-0.51% (weathered zone) 0.38-3.2% (fresh zone)	Where sulphur speciation testwork was available (fresh zone waste rock, and ore), sulphide was the dominant form of sulphur, accounting for between 94-99% of the total sulphur content.
Acid neutralising capacity, kgH_2SO_4 /t	280-540 (Hanging wall mafic tuff) 1.9-6.7 (other lithologies in the weathered zone) 25-53 (other lithologies in the fresh zone)	<0.5-380 <0.5-160 (sediment -hosted ore)	The readily available ANC[4] of the Hanging wall Mafic Tuff from the weathered and fresh zones were assessed to range up to 50 kgH2SO4/t and 20 kgH2SO4/t, respectively.
Net acid producing potential (NAPP), kgH_2SO_4/t	-539 to -23 (Hanging wall mafic tuff) Close to zero (Footwall sediment and sandstone)	-350 to -1 (most samples) 8.4 (Graphitic Sediment Hosted Massive Quartz sample)	
Net acid generation (NAG) testing NAG pH NAG acidity, kgH_2SO_4/t	2.5-9.3 <0.5 - 36	7.9-10.4 <0.5	5 of 6 waste rock samples and 9 mineralization samples generated no acidity. One waste rock sample (Footwall Sediment) was acid generating (NAG pH 2.5; NAG acidity, 36 kgH2SO4/t)
Geochemical Classification NPR AMIRA	NAF (26), UC (5), PAF (2) NAF (4), UC(NAF) (1), PAF (1 – footwall sediments)	NAF (7), UC (4), PAF (1) NAF (9), UC(NAF) 1	

[4] Readily available ANC was assessed using a modified ANC test – details not of testwork method not specified.

20.4 Mineralized Waste Management

It is understood that there is an existing stockpile comprising approximately 300,000 m^3 oxidised, non-mineralized, and waste rock. Any future project is likely to generate additional stockpiles that would need to be managed.

Geochemical characterisation datasets are limited. There is insufficient information to determine the potential for acid, metalliferous drainage from the wastes with any degree of confidence. While oxidised material is currently considered to be non-acid forming (NAF); further data are required to verify that this would be true of oxidised volumes to be mined as part of current plans. Even if classed as NAF, the potential for leaching of elements such as arsenic, antimony and selenium under pH neutral, oxidising conditions should be evaluated.

Some important issues with respect to waste rock management requirements are:

Determination of the volumes of NAF and PAF-classed materials that would require management on the surface.

For high sulphide wastes (whether PAF or NAF-classed), waste storage facilities should be designed so that potential base seepage and surface runoff are managed. This would mitigate the potential for water quality impacts from acidic, metalliferous drainage (PAF wastes) or neutral saline drainage (NAF-classed sulfidic wastes).

Should material be identified that reacts at a very high rate (e.g. high sulphide content material), then it may be advisable to use dump designs and construction methodologies that control and minimise oxidation rates. The objective would be to minimise accumulation of reaction products during storage on the surface. Such products could dissolve in contact waters when stored material is returned underground and is inundated as groundwater rebounds. Minimisation of accumulation rates on the surface will minimise potential impacts on groundwater quality later.

21 Capital and Operating Costs

21.1 Summary

The costs for the project, described in the following section have been derived from a variety of sources, including data provided by Newmarket Gold based on its existing Northern territory operations and SRK' s internal data base.

The open pit mining operating costs have been developed from, unit costs provided by Newmarket Gold, sourced from local contractors and include allowances for Mining, Drill and Blast, grade control, Indirects and road maintenance costs.

The underground mining operating costs are based on unit rates from the existing contracts for Newmarket Gold' s Northern Territory operations and estimates by SRK.

The key drivers are development AUD/m rates, drill and blast AUD/m rates, ground support costs, truck haulage AUD/tkm rates, backfill rates for cemented and waste rock fill. Assumptions have been made for supervision, underground services and mining G&A costs sourced from SRK' s internal database.

The basis of the capital costs estimates for the Process Plant and Infrastructure are presented in Section 17 and described in further detail in Section 21.2.

The mining capital cost estimates are predominantly developed of the operating costs input assumptions described in Section 21.

All cost estimates are provided in 2015 Australian dollars. Escalation, taxes, import duties and custom fees have been excluded from the cost estimates.

The capital and operating cost estimates have been consolidated into the following files:

CGC001_Maud Creek TEM_Stand- alone_Rev6.xlsx;

CGC001_Maud Creek TEM_UR_ Rev6.xlsx; and

Maud Creek TEM_ Option Summary_Rev6.xlsx.

21.1.1 Capital cost Summary

Table 21-1 summarises the total capital cost estimate for the Maud Creek Gold Project.

Table 21-1: Maud Creek Gold Project – Capital Cost Estimate

Item	Total (AUD M)
Process Plant	24.9
Surface Infrastructure	2.6
Surface Water Management	7.3
Access Road Upgrade	2.2
Open pit Mobilisation	2.0
Total-Plant and Equipment	**39.1**
Capital Development	16.4
Total Capital Cost	**55.5**

21.1.2 Operating cost Summary

The operating cost estimates applied in this Technical Report are summarised in Table 21-2 and described further in the following sections.

Table 21-2: Maud Creek Gold Project - Operating Cost Summary

	Cost (AUD M)	Cost (AUD/t)	Cost (AUD/oz rec)
Open pit Mining	29	5 /t mined	328
Underground Mining	167	55 /t mined	410
Processing	125	32 /t milled	252
Concentrate transport (AUD234/t con)	66	17 /t milled	134
Site G&A, Indirects	19	5 /t milled	39
Total Operating cost	**408**	**105 /t milled**	**822**

21.2 Processing Plant Capital Cost

The Processing Plant capital cost estimate for this PEA has been generated from designs and flowsheets described in Section 17 to a preliminary level of accuracy, i.e. +/-30%.

A standard methodology was used for estimating the preliminary capital costs. Processing flowsheets for a stand-alone plant were developed, the process design criteria generated and the flowsheet modelled using the SysCAD process simulation software. The model incorporates equipment sizing calculations developed by Simulus Engineers. These equipment sizes generate a mechanical equipment list which is then costed using a database of recent equipment costs. Additional budget pricing will be obtained from vendors at the next stage of the study. The following process areas listed below were then removed from the capital cost estimate for the Union Reefs treatment option as they exist under the new base case:

Crushing

Milling (minor allowance only for hopper and pumping)

Gravity concentration & gold room

Tailings thickening

Tailings storage

Some reagents

Raw water

Water treatment

Potable water

Gland water

Fire water

Instrument air.

Costs for the other major engineering disciplines such as earthworks, civil, structural, piping, instruments, electrical, site buildings, first fills, laboratory and IT and communications were based on typical industry factors against the mechanical equipment cost. Other factored direct capital cost items include first fills, critical spares and warehouse inventory, laboratory equipment and supplies. All factored costs were adjusted to account for the existing site, systems, cleared areas, etc. Factors were also used for freight, commissioning and vendor support, temporary facilities, mobilisation, demobilisation and EPCM. Other factored items already existing at Union Reefs such as site buildings, IT, communications and network equipment have been completely removed from the cost estimate. Finally, a contingency of 10% was applied due to the preliminary nature of the study to account for other costs not considered.

The capital cost estimate will be refined as part of future study work, with a focus on equipment sizing and costing and integration into the existing processing Union Reefs facility. It is likely there is still some double up of costs because of the preliminary level of the assessment and that the estimate can be reduced further. Examples of potential savings include the assumed requirement for a full separate process water system and full LP and plant air system. Some of the factors used for non-mechanical disciplines are also likely to be lower for a Brownfield site. They have been adjust down but could come down further with a more detailed assessment. Inevitably, other minor modification costs will creep into a Brownfield Project.

At this stage of costing, the equipment sizing is automatically calculated and pricing automatically generated from the process simulation model outputs. Sizing is at a preliminary level of accuracy only. A modular approach will be taken to the design and construction of the processing facility to reduce capital cost, schedule and site construction time.

21.2.1 Infrastructure

The capital cost estimate for the non-mining infrastructure has been generated at a preliminary level of accuracy only. The haul road upgrade costs from the Maud Creek mine site to the Stuart Highway are based on typical unit rates for widening and upgrading the existing access road. This was then benchmarked against SRK database costs for the actual construction costs of a new road for a Northern Territory Project based on first principals. The cost of the 10.7km haul road to Ross Road is AUD2.2M. No costs have been allowed for any potential upgrade of the Ross Road and Que Ping Road intersections with the Stuart Highway. These intersections will be assessed further should the Project proceed to the next stage. An additional 20% contingency has been applied to the road upgrade due to the preliminary level of engineering used in the estimate. These infrastructure capital costs are shown below in Table 21-3.

It is assumed that no additional site buildings are required at Union Reef. It is also assumed that an accommodation village is not required, with the mine workforce instead using accommodation in Katherine and other regional towns. The additional Processing Plant numbers will be accommodated at the Cosmo Village or residentially. Rail, airport and other facilities are available at Katherine. Hydrology, bore field and tailings storage facility costs are discussed separately. Power is already available for processing at Union Reefs, mine site power will be provided by IPP diesel generators sets at negligible cost to Newmarket Gold.

Table 21-3: Infrastructure Capital Cost Estimate Summary

Key Infrastructure	Factor (%)	Total cost (AUDM)
Road Upgrades		2.2
Sub total		2,.2
Contingency (20%)	20	0.44
Total		2.64

21.2.2 Duties, taxes and insurances

Goods and Services Tax (GST) and insurance has been excluded from the estimate. The Processing Plant insurance cost is already paid by Union Reefs although there may be a small premium increase to cover the new area of the plant. Mining insurance has been excluded at this stage of the Study. Construction insurance for the plant and infrastructure modifications has not been captured in the capital cost estimate.

21.2.3 Qualifications and assumptions

The capital costs are based on the assumptions above. They will be finalised with award of the work and any refinement of scope.

21.2.4 Exclusions

The following items are specifically excluded from the capital cost estimate:

- GST
- Exchange rate variations
- Additional environmental requirements beyond the scope
- Land acquisition costs
- Cost of handling and disposal of contaminated products
- Escalation
- Owners costs
- Project holding costs.

21.3 Operating Costs

The Maud Creek Gold Project operating costs have been produced for the key cost centres of geology, mining, processing, concentrate transport and general and administrative (G&A) costs. The geology and mining costs were generated by SRK and consolidated into a single 'Mining' cost. The processing and G&A costs were built up by Simulus from first principles. The concentrate transport costs were provided by Bertling and Northline.

This section summarises the operating cost estimates and the basis for them. These costs apply to this PEA. Whilst they have a Project specific basis, they are at a preliminary level of costing only and should be considered to have an accuracy in the order of ±30%. This will be refined should the Project progress to the next phase of study.

21.3.1 Mining operating costs

The mining operating cost estimate has been derived from unit rates provided by Newmarket Gold based on its existing NT operations and estimates of fixed costs sourced from SRK' s internal database. It has been assumed that contract mining will be utilised for the open pit and underground mining operations.

The following unit rates were applied to the scheduled mining physicals:

Open pit mining

- Load and Haul cost per tonne ore and waste;
- Drill and blast cost;
- Grade control cost; and
- Indirect costs (technical and Administration).

Underground Mining

- Development cost per meter;

Underground drilling and blasting costs per metre;

Underground haulage cost per tonne kilometre;

Ground support costs; and

Backfill costs.

The underground mining development costs were applied to determine the decline development cost that has been included as capital development. As such the capital development cost estimate is based on the operating cost assumptions.

The following fixed, time based (quarterly), costs have been applied to the production profile and were adjusted for the ramping up and down of the production profile. As is discussed in Section 21.4, in future stages of study the opportunity to re-allocate a portion of these costs to "capital allocation" could be considered.

Supervision - to allow for operational supervision and management;

Underground Services - to allow for the provision of additional services and activities not included in the unit rates.

Mining General and Administration - to allow for site based management and support and technical services and support including geology and grade control.

These costs are summarised in Table 21-4.

Table 21-4: Mining Operating Costs

	Cost (AUDM)	Cost (AUD/t mined)
Open Pit Mining	29	5 /t (ore and waste)
Underground Mining (Direct variable/ unit costs)	87	29
Mining Supervision	9	3
Underground Services	25	8
Mining General and Administration/ Technical Support	47	15
Capitalised development derived from operating cost inputs	16	5

21.3.2 Processing operating cost

Process operating costs have been developed from first principles using the Maud Creek stand-alone facility operating cost model. This was then modified to remove any double up in costs already incurred at Union Reefs. Reagent consumption and grinding media rates are taken from the mass balance outputs and testwork. Unit costs are based on supplier quotes provided to Simulus Engineers by vendors during the previous 6 months (not current contracts at Union Reefs which will be updated at the next phase of study). Labour requirements have been estimated based on the existing Union Reefs organisational structure assuming an extra shift of operators and maintenance is required as well as an extra flotation team member per shift. It uses recent industry assessment for salaries but will be updated at the next phase of study to use the actual salary structure at Union Reefs. Power consumption has been built up from the equipment list and power price based on the local grid prices. The bulk of the fixed costs have been removed as they are largely accounted for already through the Union Reefs fixed operating costs. The breakdown of the processing and G&A costs are provided below in Table 21-5 and Table 21-6.

The processing operating costs, including maintenance costs are estimated to be AUD18.13/t. This includes a 5% contingency. They were calculated on a throughput of 500 ktpa but through an ongoing Union Reefs Processing Plant already processing ore at an average cost of AUD28.54/t in 2015 for a throughput of 725,002 tonnes. A future review may choose to share the fixed cost savings benefits between the Cosmo underground ores and Maud Creek mineralization.

Power was based on the mechanical equipment list, loads and utilisation factors and network and electricity supply tariffs have been provided at an average unit power cost of AUD0.2177/kWh provided by PWC. The actual electrical power unit cost at Union Reefs will be updated during the next phase of study.

Labour costs are assumed to be largely absorbed by the existing Union Reefs staffing, with one additional flotation technician required for the new flotation circuit for each shift. Currently the Union Reefs Mill operates on a 9 days on 5 days off roster so another shift of operators and shift maintenance has been allowed for a 24/7 operation, an overall increase of 11 people. This also supports the concentrate bagging function. The professional and administrative roles are already filled by Union Reefs' employees and no additional allowance has been made. It remains a lean operating structure. Salaries and wages for the additional personnel are based on a McDonald Gold & General Mining Industries Remuneration Report (Australia). It used the 25th percentile salaries based on the current depressed Australian resources sector. It does not use the existing salary structure at union Reefs. This can be updated at the next stage of Study. Mining and geology manning are addressed in previous sections.

Maintenance and consumable costs including wear liners were based on typical costs from other small scale flotation circuits but applied pro-rata based on the percentage of throughput through the Union Reefs Processing Plant.

Accommodation and messing costs are based on the Cosmo Accommodation Village (200 person capacity) and assuming a local contingent to the extra operating positions. It assumes a drive-in/ drive-out operation of the new starters from Darwin or daily commuting from local towns.

A separate operating cost allowance has been made for haul road maintenance of AUD320,000/year (AUD0.64/t) . The main assumptions are provided below.

- Treatment rate of 500 ktpa on a 100% fresh, (will be conservative for the oxide and transitional component of the feed);
- Labour costs are covered by Union Reefs existing headcount except for one additional operations shift and one additional operator per shift;
- Processing costs are from the ROM pad (the ROM bin front end loader fuel is included);
- General administration costs are covered within the existing Union Reefs operating budget;
- Additional consumables costs are assigned pro- rata for the feed to plant
- An additional AUD100,000/year in freight costs is applied;
- Power is provided to Union Reefs from the grid at an average unit power cost of AUD0.2177/kWh;
- Average power consumption is 38.5 kWh/t ROM feed;
- Reagent costs include delivery to site and are based on recent budget quotes from Redox and Simulus database costs;
- Reagent consumptions are based on metallurgical testwork and process simulation;
- Water is supplied by the existing Union Reefs infrastructure;
- The bulk of the plant control samples are tested on site;
- A contingency of 5% has also been provided based on the preliminary level accuracy of the study; and
- The estimated costs are direct costs only. Financing and depreciation costs, depletion of

mineral property, interest and tax, royalties and other such indirect costs are not included.

Table 21-5: Maud Creek Gold Processing Operating Cost Summary

	Cost (AUDM)	Cost (AUD/t milled)
Processing (5% contingency)	8.74	18.13
Processing General Administration (5% contingency)	0.18	0.36
ROM Haulage to Union Reef	7.20	14.40
Haul Road Maintenance	0.32	0.64
Concentrate Transport	8.55	17.11
Total	**29.32**	**58.65**

Table 21-6: Processing and G&A Operating Costs

Processing and G&A Operating costs	**Unit**	**Cost**
Labour (excluding mining and geology)	AUDM	1.4
Reagents and grinding media	AUDM	1.9
Power (excluding mining)	AUDM	4.2
Consumables	AUDM	1.2
General and Admin	AUDM	0.2
Total	AUDM	**8.8**
Throughput	tpa	500,000
Labour	AUD/t	2.84
Reagents	AUD/t	3.72
Power	AUD/t	8.38
Consumables (including lab)	AUD/t	2.32
General and Administrative (excl. labour)	AUD/t	**0.34**
Total (Processing)	AUD/t	**17.26**
Total (Processing +5% contingency)	**AUD/t**	**18.13**
Total (GA)	AUD/t	**0.34**
Total (Processing and GA)	AUD/t	**17.60**
Contingency (+5%)	AUD/t	0.88
Total Processing & GA including contingency	**AUD/t**	**18.48**
Haul Road Maintenance	**AUD/t**	**0.64**

Union Reef' s processing operating cost for the 2015 calendar year was AUD28.15/t for 725 kt of dry ore milled, but this includes AUD2.98/t for sodium cyanide and AUD1.29/t for liquid oxygen which isn' t required to treat the Maud Creek mineralization. If an assumption is made that the fixed costs are approximately 35 – 40% at Union Reefs and are therefore removed, the additional cost associated with processing of the Maud Creek mineralization correlates closely.

The process operating cost was also compared against a number of other small flotation plants throughout Australia in the SRK database to provide further benchmarking and validation of the costs. They benchmarked well.

21.3.3 Processing General and Administrative

All G&A costs except for the RoM pad loader diesel are assumed to be fixed costs for Union Reefs and are thus not included in the estimate. This generates the G&A cost of AUD0.34/t including contingency.

21.3.4 ROM Haulage to Union Reef

Ore haulage costs have been benchmarked against similar projects and a rate of AUD0.10 /t/km has been used. This also matches the ore haulage cost from the Cosmo Underground Mine to the Union Reefs Processing Plant. This generates the ROM haulage cost of AUD14.40/t. No contingency has been applied as there is confidence in the haulage contractor cost used.

Haul road maintenance costs include the cost of dry grading, washing delineators and signage, minor repairs and re-sheeting of the road every two years. A typical rate of AUD30,000 /km/year has been applied to the 10.7km haul road from the Maud Creek mine site to Ross Road a total of AUD320,000/year. This generates the haul road maintenance cost of AUD0.64/t. No contingency has been applied.

21.3.5 Concentrate Transport

A concentrate transport cost of AUD234/t concentrate has been developed for the study. It allows for road transport of sea containers from Union Reefs to the Darwin Port, port fees, and shipping to China. This translates to AUD17.11/t of feed processed through the plant.

Preliminary concentrate transportation costs are based on an integrated transport solution supplied by a third party freight transport company. It includes all land transport, shipping, handling and storage. The basis of this costing is provided in Section 19.1 Concentrate Transport. It assumes 36,500 tpa of dry concentrate is transported at 10% moisture in 1.5 tonne bulk bags which are in turn stored in 20' shipping containers.

The cost of purchasing or leasing the 20 foot containers and variable surcharges such as the bunker adjustment factor (BAF) are not included in the transport logistics cost but the bulk of the typical fees such as the port service charge, port security fee, bill loading fee, export declaration fee and PRA lodgement are. The sea freight quote does not allow for the Seaway Bill, insurance, duty payable to customs, GST payable to customs, disbursements payable to customs & any additional quarantine surcharges on arrival, an allowance for fuel surcharge fluctuations or currency uplift. Costs were requested from FH Bertling and Northline and a summary of the budget costs are shown in Table 21-7.

It excludes hazardous goods transport and capacity has not been confirmed (not expected to be an issue at the proposed tonnages).

Table 21-7: Market Costs for Concentrate Transport

	Units	Northline	FH Bertling
Transport from site to Darwin Port			
Cost per container	AUD/container	1,752	1,685
Cost per ton	AUD/t	87.6	84.3
Cost per dry ton	AUD/dmt	97	94
Transport from Darwin to Dalian			
Cost per container	AUD/container	2,838	2,529
Cost per ton	AUD/t	142	126
Cost per dry ton	AUD/dmt	158	140
Transport from Site to Dalian	**AUD/dmt (conc.)**	**255**	**234**
Transport from Site to Dalian	**AUD/dmt (feed)**	**18.62**	**17.11**

The concentrate charges for transport from the Union Reefs Processing Plant to Darwin Port using a road transport has been estimated at AUD94 per dry metric tonne (dmt).

The shipping charges for transport from Darwin Port to Dalian Port have been estimated at AUD140 per dry metric tonne (dmt). The price excludes the variable bunker surcharge (BAF) which is applied at the time of shipment.

The flotation gold concentrate is loaded into 1.5 t capacity bulk bags, which are then loaded into shipping containers ready for transportation. The containers are loaded onto a road train that transports the concentrate by road from the Union Reefs Processing Plant to the Darwin Port. Railway transport has not been considered due to the need for a siding, capital cost, the relatively small volume of concentrate, short LoM and proximity and connectivity to Darwin via a major highway.

The average payload of each container is approximately 20 wet tonnes of concentrate. An additional bag should be loadable but this maintains a level of conservatism in the estimates. A summary of the transport logistics assumptions are shown in Table 21-8. They have been used to estimate the cost of transport.

Table 21-8: Concentrate Transport Logistics

	Units	Value
Plant Operation		
Annual production	t/y	500,000
Operating weeks per year	w/y	52
Recovery to concentrate	%	7.3
Concentrate production	t/y	36,500
Concentrate moisture content	%	10
Container Transport		
Product per container (wet)	t	20
Product per container (dry)	dmt	18
Containers per week	containers/w	48
Containers per year	containers/w	2,028

The charge for the integrated concentrate transport solution has been estimated at AUD234 per dry metric tonne (dmt). This value correlates well when benchmarked against other small Australian concentrate producers exporting containerised product to China.

At Darwin Port, the containers are stacked until required for ship loading with fortnightly sailings scheduled. Preliminary costs assume Dalian Port as the destination port in China and have a transit time of 29-36 days via Shanghai or Kaoshiung.

The cost of transporting the gold concentrate is included in the G&A cost of AUD0.34/t including contingency.

At the next level of design, logistic cost details would be further developed. The study would investigate whether individual components of the logistics chain could be separated in order to improve the overall cost. It is also possible that with detailed negotiations between the owner and logistics companies may also reduce costs once production was imminent. The next phase of study will also look at whether an additional one or two bags of concentrate can be loaded in each container to further reduce the cost per tonne.

21.4 Sustaining Capital

21.4.1 Mining Operations

No mining equipment replacement/ sustaining capital have been included as the open pit and underground mining is undertaken by contractors.

The sustaining capital allowance in the PEA provides for ongoing capital development after Quarter 2 on the production profile. On the basis of contractor mining, SRK considered that the sustaining capital items would largely be worn by the contractor.

The fixed costs associated with underground mining, including Supervision, Underground Services and Site Mining General and Administration costs have been included in the operating cost estimate as a fixed, $/Qtr estimate. These allow for the fixed cost component associated with the mining costs. In future studies some of that may be capitalised but in the opinion of SRK these are seen as small items that were considered not significant to extract as a separate line item at this stage.

At the next level of detail in future studies there is an opportunity to refine the allocation of operating and capital cost but for the level of accuracy in this study this has not been completed. SRK estimates that approximately AUD6M could be re-allocated from operating to capital cost classification.

21.4.2 Processing Plant

Maud Creek processing has been considered as an incremental cost to Union Reefs, as such the processing sustaining capital cost (and TSF works) are included in the current Union Reefs costs and have not been double counted in the Maud Creek model. There may be some minor sustaining costs required for the floatation circuit but these have not been included in the PEA at this stage and are seen as only minor.

In addition to the capital costs, economic modelling would normally incorporate a sustaining capital cost for the Processing Plant. This has been removed at this stage of study because processing is through the existing Union Reefs Plant, which has its own sustaining capital allowance. SRK notes that a small allowance may need to be included at the next phase of the Study for any early modifications, debottlenecking and other requirements as the project is ramped up.

This sustaining capital cost excludes future tailings dam lifts. It is understood that there is up to 20 years of existing tailings storage capacity at Union Reefs, another significant benefit.

21.5 Concentrate Selling Expenses

The concentrate selling costs are based on indicative terms for the China direct smelting option which makes up the bulk of the gold ounces produced. A portion of the gold is also produced as a gravity gold concentrate and will likely be intensive leached, electrowon and smelted into doré on site at Union Reefs using the existing equipment. Consideration will need to be made in whether the gravity gold can be intensively leached or whether it will need to be upgraded using shaking tables before directing smelting or whether it will require specialised smelting and refining at an established Australian gold refiner.

For financial modelling purposes, the following terms were used as provided by Shangdong Zhong Guo (China Gold Shandong) for processing at their Yantai Gold Smelter.

Concentrate (Direct smelt in China)

35,500 tpa concentrate (7.3% mass pull x 500,000 tpa)

45 g/t gold

Approximately 52,808 oz/year

Payment for 95% of contained gold in the concentrate

USD10/t processing / refining charge

Gravity concentrate into bullion (AGR or equivalent in Australia)

14,802 oz/year

AUD1 oz refinery charge

AUD5k/month for bullion transfer

Both gravity and flotation concentrate terms will be developed further at the next phase of the study.

22 Economic Analysis

22.1 Introduction

The Maud Creek Gold Project financial model was developed by SRK based on the production schedule and assumptions described in the earlier sections. All costs are constant in 2015 AUD with no provision for inflation or escalation.

The annual cash flow projections were estimated over the project life based on capital expenditures, operating costs and revenue. The financial indicators examined included after-tax cash flow, net present value (NPV), internal rate of return (IRR).

It is important to note that the PEA is preliminary in nature that it includes Inferred mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and there is certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have documented economic viability.

22.2 Principal Assumptions

The principal assumptions in the mining schedule described in Section 16 of this Technical Report and the economic parameters in Section 21.

The key project economic assumptions used in preparation of the cash flow analysis have been listed in Table 22-1. SRK notes that the time of reporting the gold price was over AUD1,600 /oz.

SRK has applied a discount rate of 5% on the basis that Newmarket Gold is a Canadian based company and will source funding from US/ Canadian markets. An increase in the discount rate to 8% reduces the After Tax NPV by AUD20M.

Table 22-1: Economic Assumptions

Description	Units	Quantity
Gold Price	Gold AUD/oz	1,550
Exchange Rate	AUD:USD	0.77
Gold Price	USD/oz	1,200
Discount Rate	%	5

22.2.1 Revenue – Sales Price Assumptions

The economic model assumes that concentrate shipments are made at the end of each time period, monthly. The payables of the shipments and associated selling expenses are assumed to occur at these same time periods within the economic model.

The payable metal terms adopted in the economic model are consistent with the current sales contract terms for the gold concentrate grades and quality. It is also assumed that all gold recovered reports to the concentrate.

22.2.2 Taxes

The TEM includes Australian Government taxes of 30% and NT tax of 20%.

All tenements within the Northern Territory, Australia are subject to a Northern Territory Government Minerals Royalty in accordance with the Northern Territory Mineral Royalty Act 1982 (as amended). This royalty is calculated as 20% of the "Net Value" of mine production, where "Net Value" equals the gross revenue from the relevant production unit less the operating costs of the production unit for the year, a capital allowance on eligible capital assets expenditure, eligible exploration expenditure and additional deductions as approved by the Northern Territory Minister for Mines.

The economic analysis treats the Northern Territory Royalty as a Post-Tax cost, as is it is influenced by losses carried forward. SRK notes that Newmarket Gold NT Holdings Pty Ltd carries tax losses that have not been included in the economic analysis. Based on the integration of the Project this has the potential to further impact the value of the Project. With no “losses” carried forward into the Northern Territory Royalty calculation, the estimates value if $107 per ounce. The estimated available tax losses as at 31 December 2015, provided by Newmarket Gold is:

Income tax non-capital losses	AUD229.8M
NT Royalty Tax Net Negative Value	AUD151.0M.

22.2.3 Royalties

A summary of the Royalty payments is presented in Table 22-2.

Table 22-2: Maud Creek Project Royalties

Project	Parties Involved		Royalty Commitment	Tenements	Comments
Maud Creek	Newmarket Gold	Harmony Gold	1% NSR on Gold	MLN 1978 MCN' s 4145 - 4146 MCN' s 4149 - 4152 MCN' s 4343 - 4348 MCN' s 3839 - 3844	Maud Creek after 250,000 ounces produced
	Newmarket Gold	Mt Carrington Mines	AUD5/ounce gold produced	MCN' s 4218 - 4225	Area south of Maud Creek
	Newmarket Gold	Biddlecombe	1% NSR on all ore/metals	MCN' s 4218 - 4225 Part of EL25054	Area south of Maud Creek

A Northern Territory Build Levy of 0.1% of the cost of construction work for civil structures that form part of the land. For the purposes of this assessment this is assumed to include the process plant, surface infrastructure, tailing storage and the paste fill plant.

22.2.4 Reclamation

Possible salvage value on plant and equipment or profits from the sale of assets has not been included in the economic model. For the purposes of this assessment it has been assumed that cash flow and existing rehabilitation bonds will be used to pay for mine closure as well as any additional reclamation required. It is recommended that this assumption is revised if the study progresses to the next phase.

22.3 Economic Summary

The annual production schedule and cashflow forecasts are presented in Table 22-3. A summary of the project economics are presented in Table 22-4.

Table 22-3: Annual production and Cashflow Estimates

		Yr. 1	Yr. 2	Yr. 3	Yr. 4	Yr. 5	Yr. 6	Yr. 7	Yr. 8	Yr. 9	Yr. 10
Tonnes Milled	000 t	345	502	452	476	484	489	385	283	315	171
Head Grade	g/t	4.6	4.7	4.9	4.2	3.8	4.1	3.8	3.4	3.5	4.1
Contained Metal	koz	51	77	72	64	60	65	47	31	36	22
Capital Expenditure	AUDM	44	7	4	1	0	0	0	0	0	0
Operating Expenditure	AUDM	43	51	45	47	49	48	40	33	33	18
Net Revenue	AUDM	67	107	100	90	82	90	65	43	50	31
Cashflow (After Tax)	AUDM	-26	49	52	42	33	42	24	11	16	10

Table 22-4: Summary of PEA Results

Parameter/ Result	Units	Quantity
Gold Price	AUD/oz	1,550
Exchange Rate	AUD:USD	0.77
Gold Price	USD/oz	1,200
Mine Life	Years	9.5
Mineral Inventory	'000 t	3,911
Diluted Gold Grade	g/t	4.2
Contained Gold	koz	528
Gold Recovery (oxide/transitional)	%	85
Gold Recovery (sulphide)	%	95
LOM Recovered Gold	koz	496
Production Rate	ktpa	500
Average Annual Gold Production	koz	52
Peak Annual Gold Production	koz	70
Annual Tonnes Concentrate (Dry)	kt	30
Concentrate Grade	g/t con	45
LOM Operating Cost	AUDM	408
LOM Cash Operating Cost	AUD/oz	822
Total Operating Costs/tonne Milled	AUD/t	105
Net Revenue (less selling expenses)	AUDM	725
Pre-Production Capital cost	AUDM	42
Sustaining Capital Cost (LOM)	AUDM	14

Table 22-5: Project Economics

	Units	Union Reefs Plant Quantity
Mining		
Mine Life	years	9.5
Mineral Inventory	‘000 t	3,911
Gold Grade	g/t	4.2
Contained Gold	koz	528
Open Pit Tonnes Ore	kt	634
Open Pit Gold Grade	g/t	5.12
Open pit Contained ounces	koz	104
Total Waste Mined	‘000 t	5,000
Strip Ratio (O/P)	t:t	8
Underground Tonnes Ore	‘000 t	3,276
Underground Gold Grade	g/t	4.02
Underground Contained ounces	koz	423
Processing		
LOM Tonnes Milled	‘000 t	3,911
Production Rate	ktpa	500
Average Annual Gold production	koz	52
LOM Recovered Gold	koz	496
Metallurgical Recovery		
Transitional to Float	%	65
Transitional to Gravity	%	20
Fresh to Float	%	75
Fresh to Gravity	%	20
Annual Tonnes Concentrate (Dry)	kt	30
LOM Tonnes concentrate (Dry)	kt	285
Concentrate Grade	g/t con	45
Economics		
Gold Price	AUD/oz	1,550
Exchange Rate	AUD:USD	0.77
Gold Price	USD/oz	1,200
LOM operating cost	AUDM	408
LOM operating cost	AUD/t milled	104
LOM operating cost (Payable)	AUD/oz	856
LOM operating cost (Payable)	USD/oz	662
LOM Payable (Saleable Gold)	koz	477
Net Revenue (less selling expenses)	AUDM	725
Capital cost	AUD M	56
Pre-tax Cashflow	AUDM	261
Pre-tax NPV (5)	AUDM	201
Pre-tax IRR	%	116
Payback Period (pre-tax)	Qtr	6

	Units	Union Reefs Plant Quantity
After Tax Cash Flow	AUDM	182
After Tax NPV (5)	AUDM	137
After Tax IRR	%	80
Payback Period (After Tax)	Qtr	6

22.3.1 Sensitivity

Cash flow sensitivities (+/- 15% and 30%) to gold price, exchange rate (AUD:USD), metallurgical gold recovery, mill feed gold grade, capital costs and operating costs has been completed and presented in Figure 22-1.

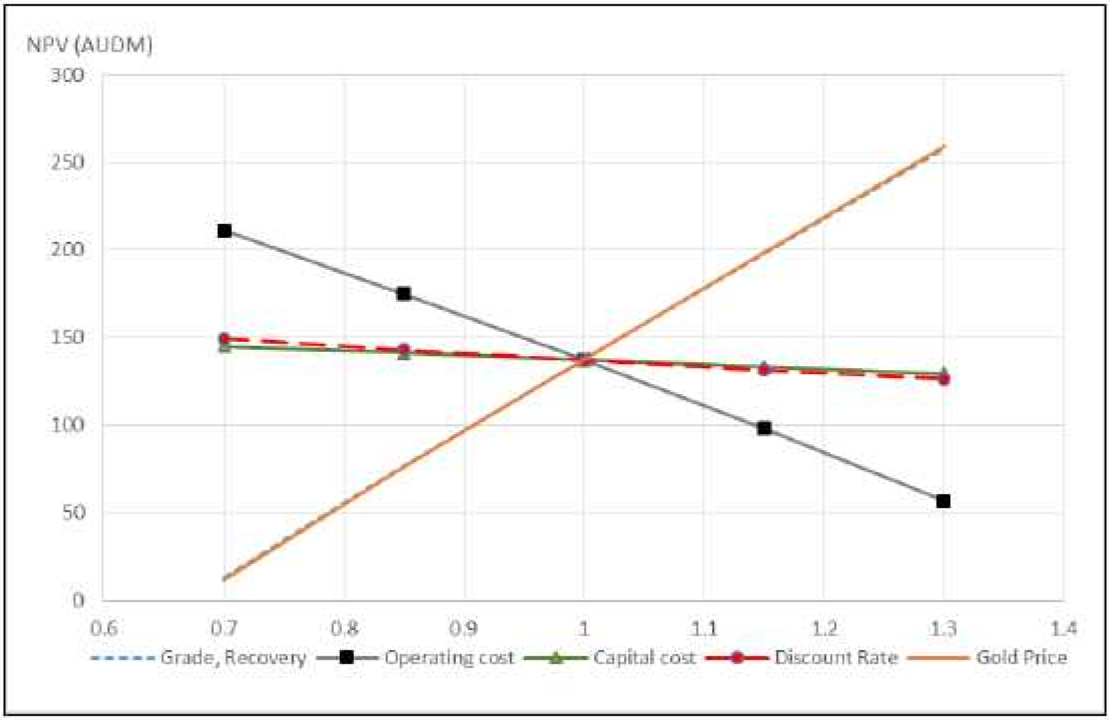

Figure 22-1: Sensitivity Analysis

The key drivers are the revenue assumptions (Grade, Recovery and Gold Price) and the operating cost assumptions.

Sensitivity to the Gold Price is presented in Table 22-6.

Table 22-6: Gold Price Sensitivity

Gold Price (AUD/oz)	1,400	1,450	1,500	1,550	1,600	1,650	1,700
Pre-Tax NPV5% (AUDM)	145	163	182	201	220	239	257
Pre-Tax IRR%	85	95	106	116	127	138	150
After-Tax NPV5% (AUDM)	98	111	124	137	150	163	177
After-Tax IRR%	59	66	73	80	87	94	102

SRK notes that Capital Cost estimates used for the sensitivity analysis excludes the capital development as the cost driver for this is the unit operating costs. Impact on the capital development costs is included in the operating cost sensitivity.

A breakdown of the operating cost components, including underground capital development is presented in Figure 22-2.

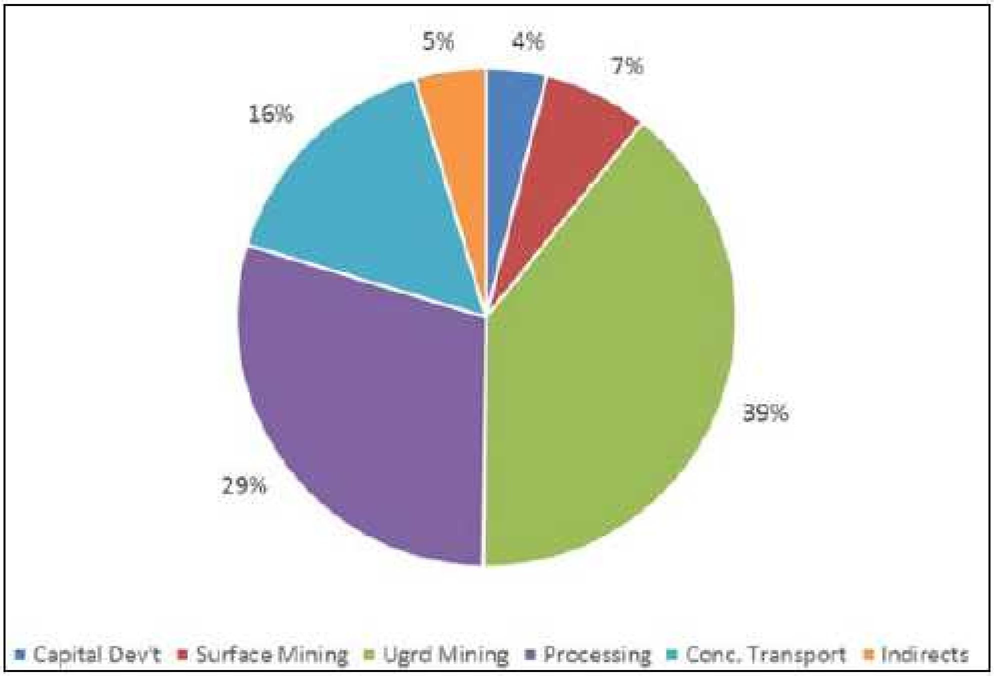

Figure 22-2: Breakdown of Impact of Operating Cost Sensitivity

23 Adjacent Properties

Other projects in the region managed and owned by Newmarket Gold include:

The Pine Creek Gold Project (Figure 23- 1), located approximately 190 km south southeast of Darwin accessible by the Stuart Highway, directly to the west of the Township of Pine Creek.

The Union Reefs Gold Project (Figure 23- 2), located approximately 180km south- southeast of Darwin accessible by the Stuart Highway, directly to the north of the Township of Pine Creek.

The Burnside Gold & Base Metals Project (Figure 23-3), located approximately 130 km to the South-southeast of Darwin accessible by the Stuart Highway.

The Moline Gold and Base Metals Project (Figure 23-4), located approximately 40 km to the Northeast of the Township of Pine Creek, accessible by the Kakadu Highway.

The Yeuralba Gold and Base Metals Project (Figure 23-5), located approximately 45 km to the northeast

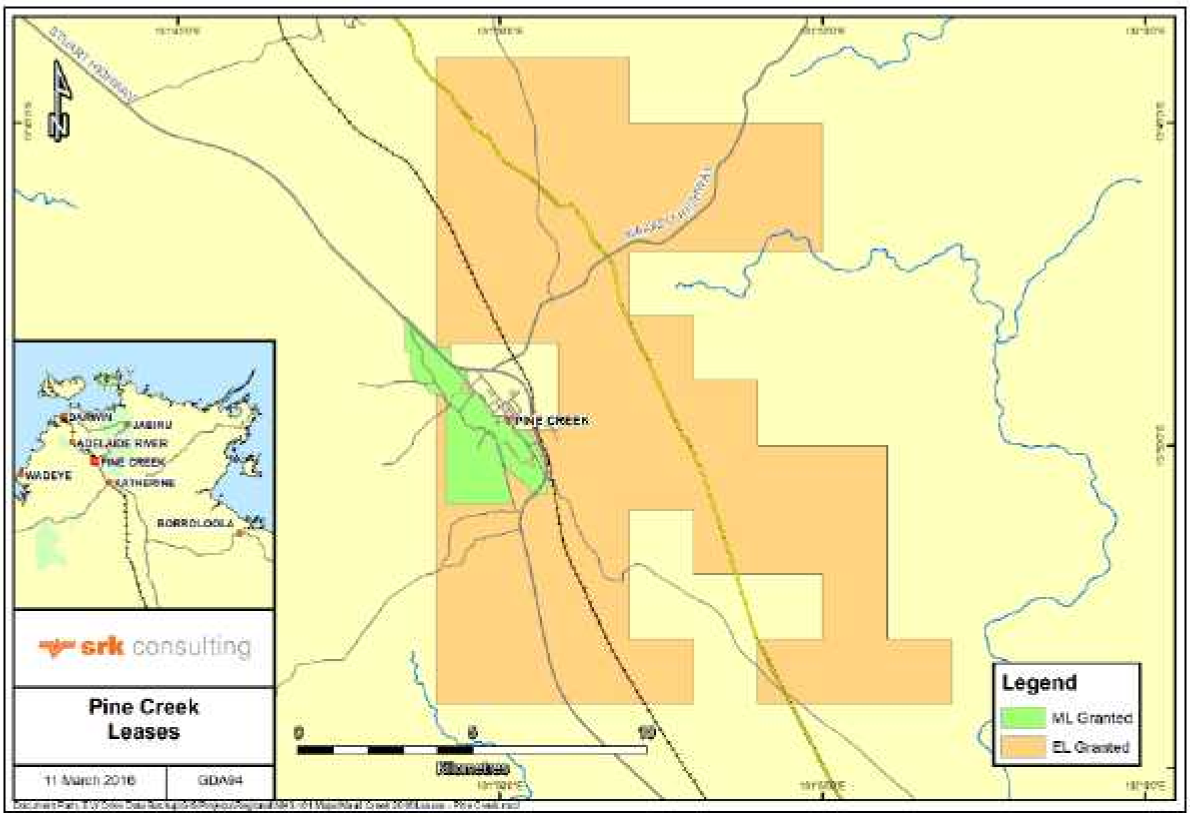

Figure 23-1: Pine Creek Gold Project Tenements

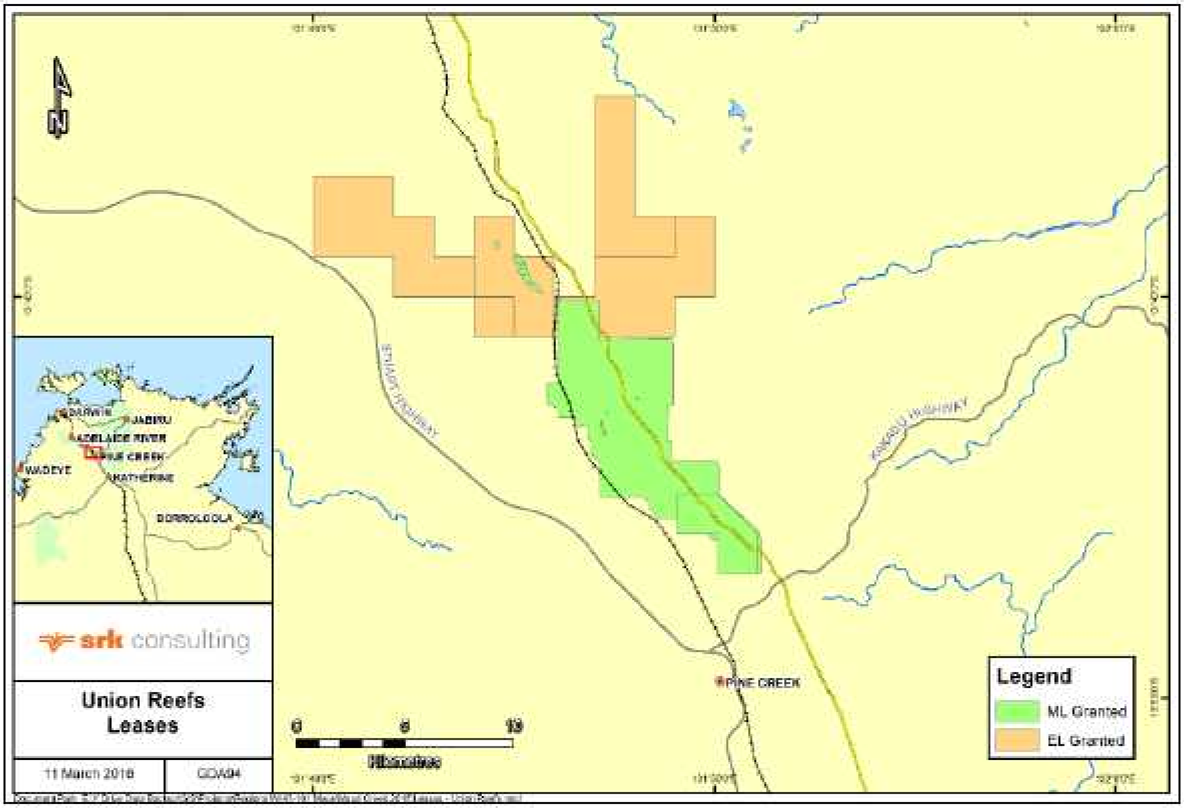

Figure 23-2: Union Reefs Gold Project Tenements

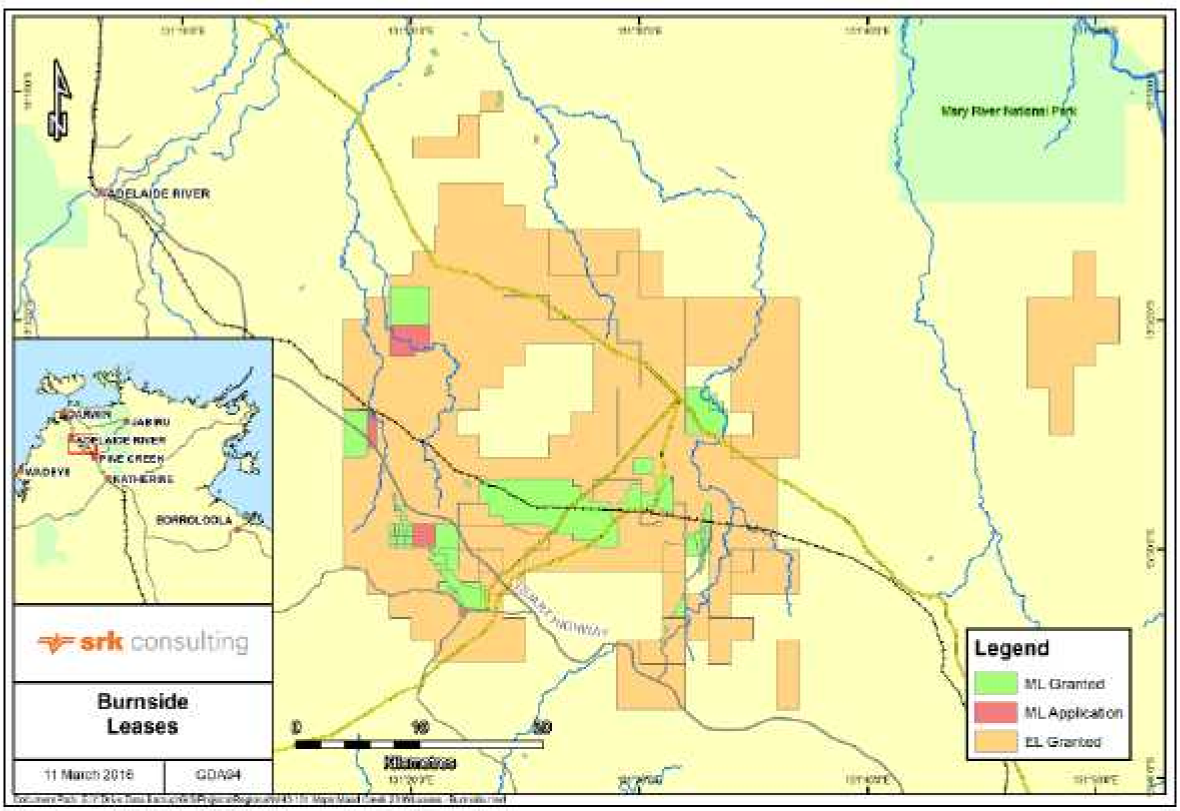

Figure 23-3: Burnside Gold Project Tenements

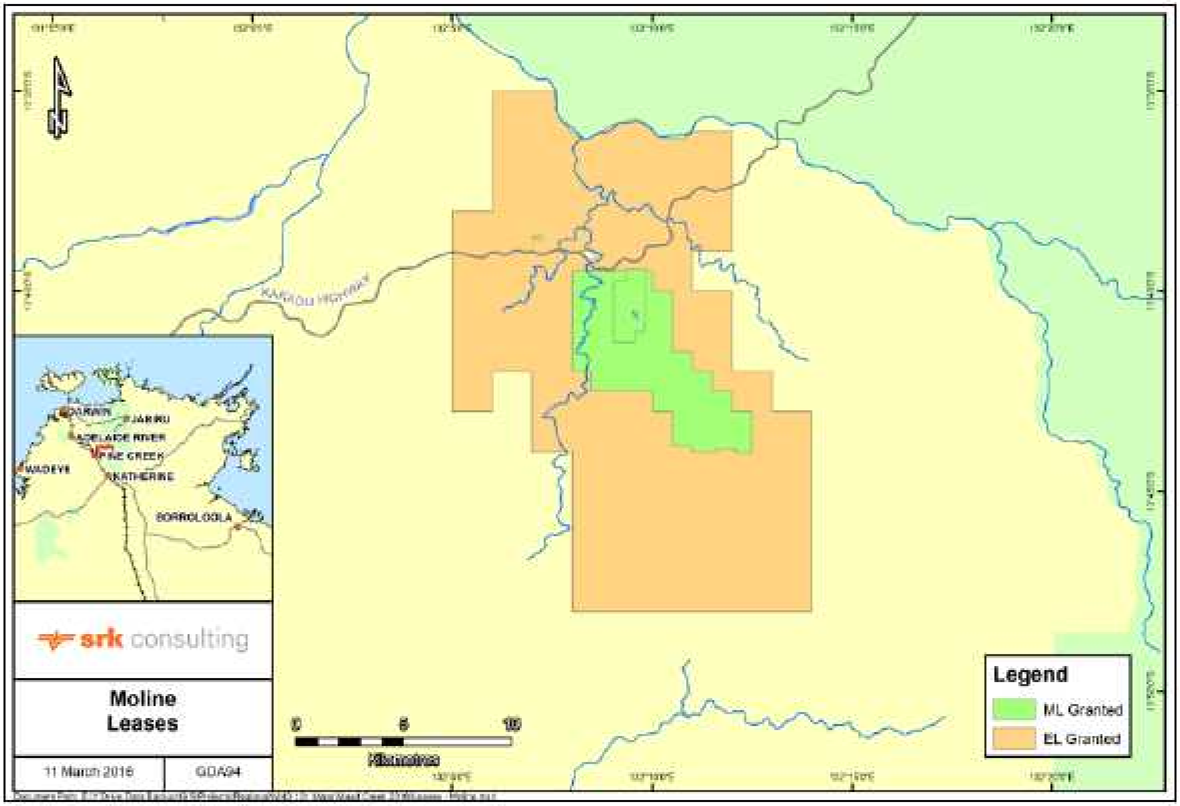

Figure 23-4: Moline Gold Project Tenements

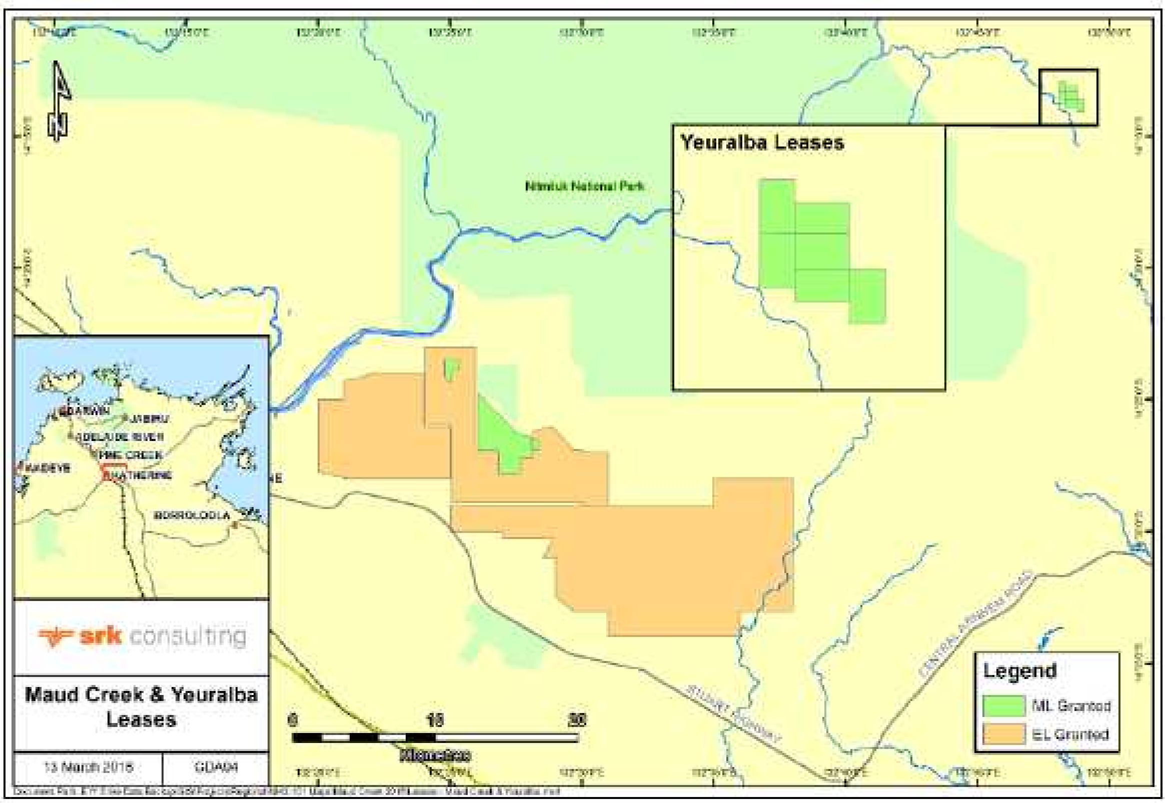

Figure 23-5: Maud Creek & Yeuralba Gold Project Tenements

24 Other Relevant Data and Information

No other relevant / material data has been excluded from the Technical Report.

25 Interpretation and Conclusions

The modelling of vein and grade volumes for the 2015 estimate takes a very different approach to the 2012 model as described in the sections above. The 2015 model incorporates a detailed structural, vein and lithological model in the construction of the various estimation domains. This was a deliberate decision as it was SRK' s understanding from discussions with Newmarket Gold and from reading previous reports that there were potential deficiencies in the very linear grade only approach previously used. Concerns had been expressed in some previous reports that insufficient attention had been paid to the geology and that the previous models may have diluted a high grade, geologically controlled core to the main zone thereby creating a model that underestimated grade and overestimated tonnages at economic cut-offs.

There are a number of differences between the 2012 and 2015 modelling approach. The 2015 model uses the following:

Pure geology to define the main and minor vein domains. The 2012 used grade only. Consequently the 2015 vein model contains considerably lower tonnage and slightly elevated grade in comparison;

Grade halos to capture both high and low grade outside the geological veins. This captures low grade material that was not modelled in 2012 which may be of value in an open pit scenario; and

Orientation controls on the grade halos derived by the combined fault / lithology contact model resulting in multiple orientations and fattening around fault and contact intersections.

The 2015 model is more robust in terms of its geological basis and this has led to a slightly higher grades but a reduction in contained gold. Only further drilling can define true connectivity of the mineralization in widely spaced areas.

A discussion on the risks and opportunities in the Mineral Resource model as discussed in Table 25-1.

Table 25-1: Mineral Resource Model Risks and Opportunities

Project Element	Economic Risk Level	Comment	Opportunity
Database - Exploration data	Low	Historical and recent data have been re-collated and re- validated for this Mineral Resource estimate.	
Assaying	Low	QAQC for recent and older assaying shows no material issues. Arsenic assaying has incomplete coverage.	Additional assaying for Arsenic may be beneficial depending on the processing method.
Surveying	Low	Both collar surveys and downhole surveys completed to a high level of accuracy for recent drilling. Representative collars resurveyed for older drilling with no significant discrepancies.	
Geology	Low	Detailed logging and interpretation together with evidence from both regional structural features and detailed in pit mapping informs the geological understanding	Additional drilling may be able to add detail to the interaction of structures controlling mineralization at depth.

Project Element	Economic Risk Level	Comment	Opportunity
Geological modelling	Low	A detailed structural and lithological model has been built and incorporated into the estimation domain construction.	Additional drilling may be able to add detail to the interaction of structures controlling mineralization at depth.
Resource Estimation	Low	Ordinary kriging cross checked and validated with theoretical grade tonnage curves and alternative search parameters has been used.	The project may benefit from simulation studies or non-linear estimates if detailed studies at selective mining unit block sizes are required in the future.

The absence of suitable data has led to low-confidence in the geotechnical conditions. Additional data will be required to improve confidence and refine decisions on mining methods and the mine design. The mining method studies are linked to the decision on the location of the Processing Plant and the availability of pastefill.

The Maud Creek mineralization has been subject to extensive metallurgical testing. However there remain some gaps that should be covered for future studies that could reduce the future design risks if it was decided to undertake additional testwork earlier. This would include, lithological domain characteristics, SAG milling parameters, testing samples from depth, flotation testwork and concentrate production and specification.

PEA Results

It is important to note that the PEA is preliminary in nature that it includes Inferred mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and there is certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have documented economic viability.

Based on the economic findings of the PEA, SRK concludes that the Project has merit and that Newmarket Gold consider progressing the study to the next level of detail.

In doing so SRK provides recommendations that Newmarket Gold consider progressing the additional work programs (outlined in Section 26) to better understand the key technical risk areas to support future evaluations.

26 Recommendations

Drilling Campaign

Infill drilling in the parts of the Mineral Resource currently classified as Indicated and Inferred would enable an upgrade of the Mineral Resource Classification. An approximate meterage and cost to complete this from surface down to 850 mRL is provided in Table 26-1 assuming the drilling takes place from surface. A number of sections of the geological model remain open down dip with good grades seen in the last hole down dip. Extension drilling is recommended to test these areas. Meters and costs to complete these are shown in Table 26-1 assuming drilling from surface. Costs are based on RC collars and 50m diamond drill tails.

Table 26-1: Recommended Geological drilling

Target	Current exploration status	Potential End of 2016 Status	Description	Drilling (m)
Infill Drilling	Indicated and/or Inferred	Measured and/or Indicated	Increase confidence in estimated Mineral Resource	9,200
Extension drilling	Down dip or along strike from current Mineral Resource	Indicated and/or Inferred	Close off or extend Mineral Resource volumes	2,200

Geotechnical

If an additional phase of geotechnical work is undertaken, SRK recommends that a drilling program be considered to provide additional geotechnical data and infrastructure for hydrogeological testwork. These programs should be combined where possible to assist in any further resource definitions requirements.

Based on further drilling and geotechnical data, further studies should be undertaken to improve the confidence of the understanding of the geotechnical domains and to provide a basis for the improving the design guidelines.

Recommendations regarding phase two work, summarised in Table 26-2, includes the following:

Dedicated geotechnical drilling program to:

- Target area' s that require additional data such as the footwalls sediments, and crown pillar area;
- Drilling proposed locations of LOM infrastructure – portals, declines, vent shafts etc.;
- Improve understanding of structural characteristics including continuity and orientation variability;
- To collect representative samples for laboratory testing;

Mapping and photogrammetry in current pit;

Geotechnical laboratory testing using NATA certified laboratories;

Detailed backfill design for potential mining methods (including CRF);

Potential stress measurement testing (AE or DRA); and

Numerical modelling of proposed mining layout and sequence (including crown pillar and central pillar sequence)

Table 26-2: Geotechnical drill program targets and estimated depths

Description	Drillholes (No.)	Depth (m)	Total (m)
To investigate sediments at depth	4	600	2,400
To supplement data and sediments across mid depths of mine	4	500	2,000
Portal and boxcut investigation drill holes	3	50	150
To investigate Infrastructure located in sediments (can be separated into multiple shorter drill holes)	1	600	600
Open Pit geotechnical holes - will provide data on crown pillar area and upper sediments	6	150	900
Total meters for proposed program			**6,050**

Geometallurgy

Insufficient information was available from the metallurgical reports to create preliminary spatial domains for the physical processing parameters, such as grindability. The metallurgical report Core Process Engineering Report No. 140-001 outlined JK Drop Tests which had been conducted, resulting in a Bond ball Work Index of 18-19 kWh/tonne for the main lode within the deposit. Previous geology reports, as well as a site visit to inspect the core, gave an indication of the mineralogy of the Maud Creek deposit, which has an impact on the hardness of the mineralization. From a geometallurgical perspective, between and within each mineralized domain there will likely be a range of hardness values which will need to be established. Quartz alteration at Maud Creek has been identified as varying between the primary lode (high percentage quartz veining), moderate (footwall and hanging wall lodes stock work veining) to low (sandstone and tuff country rocks). However variations in silica content within these lodes will definitely occur, as alteration boundaries are normally pervasive across lithology boundaries. A possible way to better define the hardness parameters is the use of proxies. If silica analysis is included in any future assay testing of the mineralized zones, this can be used to identify target areas for JK drop weight tests. A regression calculation between the A*b result, Bond work indices and other comminution parameters versus the silica content can then be determined, allowing a predictive model of the hardness to be created. Mineral analysis would need to be conducted to ensure the silica content is reflecting quartz alteration which has a high hardness (Mohs scale 7) and not feldspar (Mohs scale 5-6) or mica minerals (Mohs scale 2-3) which have lower hardness and varying crystallography, resulting in different grinding behaviours. The Geology of the Maud Creek Gold Deposit and Maud Creek Reconciliation report by AngloGold in 2000 identified plagioclase laths and alkali feldspar minerals in the Andesite unit located to the south of the deposit, but not within the mineralized lodes. However, sericite (a type of muscovite) and chlorite alteration (most likely the mineral clinochlore) are both quite prevalent in all three mineralized zones, so their prevalence would need to be established as well. Similarly, the extent of the hematite and marcasite alteration noted in all three mineralized zones (present as pseudomorphs of pyrite in the oxide zone) needs to be quantified, as all three will have moderate-high hardness (6-7), but also a brittle tenacity upon breakage, which will influence the grindability.

The Maud Creek mineralization is going to be very hard and abrasive so finding a correlation with proxies such as silica, for example, is recommended by SRK. Equally, determining correlations between arsenic, sulphur and gold to help generate the flotation relationship correlations, and approximated the amount of arsenic in the final concentrate is also recommended. The arsenic and sulphur contents will also be a good indicator of recovery.

Mining

Should the project proceed SRK recommends that a range of sensitivities/ scenarios are considered at a high level to inform the project team of the implications of these constraints. The sensitivities should include combinations of considering the open pit oxide mineralization as a mill feed source with both the underground preferences retained and removed. Conventional open pit mining techniques are proposed.

The opportunity exists to modify the design to interact with the open pit design but this will potentially have constraints on production continuity.

The mine design and schedule should be revised based on the findings of the geotechnical review.

Processing & Infrastructure

The following additional recommendations are provided based on the findings of this PEA and the base case being to process mineralization at the Union Reefs Processing Plant, after modifications, rather than at a standalone processing facility:

Undertake a further review of the metallurgical testwork to understand the behaviour of the oxide mineralization which were largely excluded from the previous Phase 1 review as processing of oxides was not part of the base case.

Undertake testwork to understand the intensive leach behaviour of the gravity gold concentrate, generate flotation concentrate for customer testing and assessment, other minor testwork as discussed in the main body of this report.

Undertake a specialist comminution circuit capacity study processing Maud Creek mineralization through Union Reefs Processing Plant.

Undertake a more detailed integration study of a new flotation, dewatering and concentrate storage circuits into the Union Reefs Processing Plant in consultation with the Union Reefs and Newmarket Gold' s technical groups. This would include scope of work, opportunity upgrades at Union Reef, tie- in requirements and the campaign operating philosophy.

Update the capital cost to suit the next level of study with Union Reefs as the base case.

Update the operating costs to suit the next level of study with Union Reefs as the base case, using their power, labour, accommodation and other costs. Agreement on the share of benefits to Maud Creek and Cosmo projects is also required.

Develop cost of upgrading the access road from the Maud Creek Mine Site into Katherine.

Progress discussions with the local Government authorities to confirm in principal that haulage through Katherine and up to Union Reefs is acceptable – this is a potential fatal flaw to the base case processing option.

Confirm capital upgrade requirements attributable to Newmarket Gold (if any) to the main Stuart Hwy.

Confirm road train maximum tonnage acceptable to Local and Territory Government.

Progress flotation concentrate off-take discussions, to provide more confidence in terms and conditions.

27 References

Bremner, P. and Edwards, M. 2012. *Report on the Mineral Resource and Mineral Reserve of the Maud Creek Gold Project*.

Pakalnis, R C. Poulin, R. and Hadjigeorgiou, J. 1995. *Quantifying the cost of dilution in underground mines,* Mining Engineering, 47(12): 1136-1141.

Scoble, M.J., Moss, A., 1994. *Dilution in underground bulk mining: Implications for production management, Mineral Resource evaluation II, methods and case histories*, Geological Society Publication No. 79, pp. 95-108.