REVISED NI 43-101 TECHNICAL REPORT AND PRELIMINARY ECONOMIC ASSESSMENT FOR THE TOROPARU GOLD PROJECT, UPPER PURUNI RIVER REGION OF WESTERN GUYANA

PREPARED FOR: GCM MINING CORP.

REPORT DATE: FEBRUARY 4, 2022

REPORT EFFECTIVE DATE: DECEMBER 1, 2021

EFFECTIVE DATES:

Mineral Resource Estimate: November 1, 2021

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TOROPARU GOLD PROJECT
UPPER PURUNI RIVER REGION
OF WESTERN GUYANA

Prepared for:

GCM Mining Corp.

Project # 21125-01

Technical Report Effective Date: December 1, 2021

Mineral Resource Effective Date: November 1, 2021

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Technical Report Effective Date: December 1, 2021

Mineral Resource Effective Date: November 1, 2021

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The undersigned prepared this Technical Report, titled Revised NI 43-101 Technical Report and Preliminary Economic Assessment, Toroparu Gold Project, Upper Puruni River Region of Western Guyana, with an effective date of December 1, 2021 in support of the public disclosure. The format and content of this report conforms to National Instrument 43-101 of the Canadian Securities Administrators.

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FORWARD-LOOKING INFORMATION

This document includes certain “forward-looking” information and "forward-looking statements" within the meaning of applicable securities legislation, together, forward-looking statements. All statements, other than statements of historical facts constitute forward-looking statements. Forward-looking statements estimates and statements that describe the Company's future plans, objectives or goals, including words to the effect that the Company or management expects a stated condition or result to occur. Forward-looking statements may be identified by such terms as "believes", "anticipates", "expects", "estimates", "may", "will", "could", "would", "if", "yet", "potential", "undetermined", "objective", "plan" or similar expressions. Since forward-looking statements are based on assumptions and estimates and address future events and conditions, by their very nature, they involve inherent risk, and uncertainties. Although these statements are based on information currently available to the authors of this Technical Report and the Company, the authors provides no assurance that actual results will meet the expectations set forth herein. Risks, uncertainties, and other factors, known and unknown, involved with forward-looking statements could cause actual events, results, performance, prospects, and opportunities to differ materially from those expressed or implied by such forward-looking statements. Forward-looking statements in this Technical Report include, but are not limited to, the Company's objectives, goals, future plans, statements, exploration results, potential mineralization, estimation of Mineral Resources, exploration, and mine development plans, the timing of the commencement of operations and estimates of market conditions. Factors that could cause actual results to differ materially from such forward-looking statements include, but are not limited to the failure to identify Mineral Resources, failure to convert estimated Mineral Resources to reserves, the inability to complete a feasibility study which recommends a production decision, the preliminary nature of metallurgical test results, geotechnical challenges, delays in obtaining or failures to obtain required governmental, environmental, or other project approvals, political risks, uncertainties relating to epidemics, pandemics and other public health crises, including COVID-19 or similar such viruses, uncertainties relating to the availability and costs of financing needed in the future, changes in equity markets, inflation, changes in foreign currency exchange rates, fluctuations in commodity prices, delays in the development of projects, capital and operating costs varying significantly from estimates and the other risks involved in the mineral exploration and development industry, and those risks set out in the Company's public documents filed on the System for Electronic Document Analysis and Retrieval ("SEDAR"). Although the authors believes that the assumptions and factors used in preparing the forward-looking statements in this Technical Report are reasonable, undue reliance should not be placed on such forward-looking statements, which only apply as of the date of this Technical Report, and no assurance can be given that such events will occur in the disclosed time frames or at all. The authors disclaim any intention or obligation to update or revise any forward-looking statements, whether as a result of new information, future events, or otherwise, other than as required by applicable law.

CAUTIONARY NOTE TO U.S. READERS REGARDING ESTIMATES OF MEASURED, INDICATED AND INFERRED RESOURCES

This Technical Report uses the terms "Measured" and "Indicated" Mineral Resources and "Inferred" Mineral Resources. The Company advises U.S. investors that, while these terms are recognized by the U.S. Securities and Exchange Commission ("SEC") under Regulation S-K subpart 1300, there are differences between the definitions ascribed to such terms under Regulation S-K subpart 1300 and the CIM Standards.

The estimation of "Measured" and "Indicated" Mineral Resources involves greater uncertainty as to their existence and economic feasibility than the estimation of proven and probable reserves. The estimation of "Inferred" resources involves far greater uncertainty as to their existence and economic viability than the estimation of other categories of resources. It cannot be assumed that all or any part of a "Measured," "Indicated," or "Inferred" Mineral Resource will ever be upgraded to a higher category.

Under Canadian rules, estimates of "Inferred Mineral Resources" may not form the basis of feasibility studies, prefeasibility studies, or other economic studies, except in prescribed cases, such as in a preliminary economic assessment under certain circumstances. Under SEC rules, estimates of "Inferred Mineral Resources" may not be considered when assessing the economic viability of a mining project, and may not be converted to a mineral reserve. Under U.S. standards, mineralization may not be classified as a "mineral reserve" unless the determination has been made that the mineralization could be economically and legally produced or extracted at the time the reserve determination is made. U.S. investors are cautioned not to assume that any part or all of a "Measured," "Indicated," or "Inferred" Mineral Resource exists or is economically or legally mineable. Information concerning descriptions of mineralization and resources contained herein may not be comparable to information made public by U.S. companies subject to the reporting and disclosure requirements under the SEC's Regulation S-K subpart 1300.

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1 SUMMARY

Nordmin Engineering Ltd. (Nordmin) was retained by GCM Mining Corp. (GCM Mining or the Company) to prepare a Canadian National Instrument 43-101 (NI 43-101) Technical Report (Technical Report) and Preliminary Economic Assessment (PEA) for the Toroparu Gold Project (the Project). The Project is comprised of the Toroparu Deposit and the Sona Hill Deposit. It is situated in the Upper Puruni River Region of western Guyana, South America.

1.1 Terms of Reference

This Technical Report supports the disclosure of Mineral Resources for the Project in the Company news release of December 1, 2021, entitled "GCM Mining Announces Updated Mineral Resource Estimate and Positive Preliminary Economic Assessment for Its Toroparu Project in Guyana". All measurement units used in this Technical Report are metric unless otherwise noted. Currency is expressed in United States dollars (US$). The Technical Report uses Canadian English.

Mineral Resources are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (May 2014; the 2014 CIM Definition Standards) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (November 2019; 2019 CIM Best Practice Guidelines).

1.2 Geological Setting, Mineralization and Deposit Types

The Guiana Shield, the northern half of the Amazonian Craton, underlies the eastern part of Venezuela, Guyana, Suriname, French Guyana, and parts of northern Brazil. It is also among the least documented of Precambrian terranes due to thick weathering profiles, tropical vegetation, and tertiary sands (Voicu, Bardoux, & Stevenson, 2001). This region is bound in the north by the Atlantic Ocean and the south by the Amazon-Solimoes basin. There are two undisputed terranes in the Guiana Shield, the Imataca Complex in northwestern Venezuela and the Trans-Amazonian granitoid-greenstone belts in the easternmost extension in Amapá, Brazil. The Toroparu Deposit is located close to and between two major lineaments; the WNW oriented Puruni fault zone, to the southwest, and the NNW striking Wynamu fault, likely affecting the southeast portion of the deposit. Within such a regional structural pattern the mineralized zones of the Toroparu Deposit can be interpreted as east-west oriented, west plunging, dilational zones within an WNW oriented, oblique sinistral strike-slip fault zone. More structural evidence is needed to fully support this interpretation of higher-grade E-W lenses within the overall WNW oriented orebody.

The Toroparu Deposit mineralization is oriented in a west-northwest direction with cross-cutting east-west mineralized structures. The system corresponds to a 2.7 km long and 200 m to 40 m wide body and extends to over 400 m in depth. The mineralized body occurs along the northwestern boundary of a tonalitic to quartz dioritic intrusion within a series of mafic volcanics with a thick, gradational layer of saprolite material. Saprolite results from deep tropical weathering, resulting in the larger part of the original rock mineralogy being replaced by clays. Quartz veins and veinlet networks survive quite well in saprolite and contain occasional free gold grains. Sulphides tend to be completely leached and removed, leaving relic voids, and/or oxidized spots. The Toroparu Deposit sits in a topographic low and is near the Puruni and Wynamu rivers. This has resulted in the upper part of the lateritic profile being eroded. Bedrock substratum is overlain by a thin, 1 m residual soil layer, followed by a 10 m 35 m thick saprolite layer. Saprolite rock is the transitional zone between saprolite and fresh rock, creating a gradational contact several metres thick at the Toroparu Deposit.

Within the Toroparu Deposit mineralization is hosted by a paleoproterozoic greenschist facies metamorphic volcano-sedimentary (VS) sequence in contact with a tonalitic to quartz dioritic intrusives. Gold and copper mineralization appears to be largely controlled by a series of moderately developed, dilational brittle-ductile fracture veinlet stockworks. This dilational fracture veinlet stockwork forms a NW-SE trending mineralized corridor with two sets of cross-cutting higher-grade structures. Where these structures intersect, there is a large increase in the grade of both gold and copper. There is also either massive veining or vein breccias in these intersections. The main area of the Toroparu Deposit (the “Main Area”) contains the majority of known mineralization, which is still open at depth. The northwestern lens (the “Toroparu Deposit NW Area”) of the Main Area appears to have slightly lower concentrations of gold grades, but this could be due to a lack of drill density; mineralization here is also open along strike to the NW and at depth.

The Sona Hill Deposit differs from the Toroparu Deposit in the absence of potentially economic quantities of copper mineralization. Gold mineralization is hosted in sub horizontal, shallow dipping structures. It has two sets of identified cross-cutting high-grade gold structures. The Sona Hill saprolite is generally thicker, as the 25 m to 30 m of topographic hill results in a greater depth to the water table. Sap-rock and saprolite layers can reach up to 60 m thick in the Sona Hill Deposit.

Similar to the Main Area, and SE Area of the Toroparu Deposit, mineralization at the Sona Hill Deposit is mainly hosted within intrusive lithologies. These intrusives are petrographically described as porphyritic/micro-porphyritic ± equigranular granodiorite to quartz diorite. Metavolcanics are foliated andesitic volcaniclastics and intermediate to felsic flows. Quartz veining is typically white-crystalline quartz, and can be associated with feldspar, carbonate, tourmaline, sericite, and chlorite with minor sulphides (pyrite). Veins/veinlets are variable in size but generally range from 0.5 cm to 10 cm, density varies significantly. Alteration is quartz-sericite-carbonate-chlorite which is both pervasive throughout the deposit and present as vein halos.

The Toroparu and Sona Hill Deposits are a part of a single coherent structural system related to thrusting that carries hanging wall blocks eastward over footwall blocks. Mineralization here is hosted within the frontal part of the back-thrust zone.

1.3 Exploration and Drilling

Until the beginning of 2011, the Upper Puruni Concession package (1,000 km^2) had remained unexplored. A systematic surface sampling and mapping approach was implemented starting in 2011, focused primarily on geological potential for gold and/or base metals. Targets were originally selected from interpretations of airborne geophysical data and satellite imagery. In areas where geochemical sampling yielded positive results, tighter grid spacing for ground geophysics was carried out.

Geochemical samples were taken from the soil layer, and if possible, the laterite layer, averaging 0.5 m to 0.3 m depth. This was done using a hand auger. The geochemical sampling resulted in identifying the Toroparu Deposit NW area, the Ameeba hills geochemical anomaly – which led to follow up diamond drill hole (DDH) drilling – a possible extension of the Toroparu Deposit.

A geochemical sampling program in 2012 extended to areas south and southeast of the Toroparu Deposit and added 3,251 samples. Sampling during this program confirmed three new anomalies, Sona Hill, Sona Hill South, and Majuba located south and southeast of the Toroparu Deposit.

Drilling has occurred at the Project from 2006 through to 2021, directed primarily at the Main and SE Areas of the Toroparu Deposit. At the end of 2021 over 14 years, a total of 215,154 m of resource definition drilling was completed in 528 holes. Since 2013 most of the exploration drilling throughout the Project has been directed toward the Toroparu and Sona Hill deposits and the Wynamu target area. Sona

Hill was drilled from late 2015 to early 2018 with 181 diamond drill holes and 20,850 m of drilling: sufficient for resource estimation. Wynamu was drilled with 62 core holes for 6,432.6 m of drilling and further drilling is required to delineate a mineral resource.

The November 2021 Mineral Resource Estimate was prepared by Nordmin following a two-phase diamond drill program in 2020-2021 which comprised a total of 20,750 m in 114 drill holes. Previously the deposit was modelled as a large low-grade, high tonnage system. A new interpretation of the existing data identified cross-cutting high-grade structures throughout the deposit and the results of the new drilling confirmed the new resource model.

The quantity and the quality of lithological, collar, and downhole survey data collected in the various exploration programs by various operators are sufficient to support the Mineral Resource Estimate. The collected sampling is representative of gold, total copper, cyanide soluble copper, and silver data in the deposits, reflecting areas of higher, and lower grades. The analytical laboratories used for legacy and current assaying are well known in the industry, produce reliable data, are properly accredited, and are widely used within the industry.

Nordmin is not aware of any drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results. In Nordmin's opinion the drilling, core handling, logging, and sampling procedures meet or exceed industry standards, and are adequate for the purpose of Mineral Resource Estimation.

Nordmin considers the Quality Assurance (QA)/Quality Control (QC) protocols in place for the Project to be acceptable and in line with standard industry practice. Based on the data validation and the results of the standard, blank, and duplicate analyses, Nordmin is of the opinion that the assay and specific gravity (SG) databases are of sufficient quality for Mineral Resource Estimation for the Project.

1.4 Mineral Processing and Metallurgical Testing

The Company completed multiple metallurgical testwork programs from 2009 through 2020 that have produced information regarding the physical properties of the various economic material grade mineralization in the Toroparu and Sona Hill Deposits and their response to comminution, gravity concentration, rougher and cleaner flotation, cyanide leaching and ancillary processes.

For the Toroparu Deposit, testwork indicates that both "Average Copper Ore" (ACO) and "Low Copper Ore" (LCO) benefit from gravity concentration in the flowsheet. (Note that the term "Ore" as used here is a naming convention dating back to the May 2013 Prefeasibility Study (PFS) to identify two different categories of mineral processing materials and is not meant to convey positive economic connotations.)

Flotation testwork conducted to determine the amenability of LCO to flotation shows that while Cu and Au recoveries from LCO are acceptable, the relative loss in Au recovery versus a cyanide leach was not sufficiently offset by an increase in Cu flotation recovery to warrant processing of LCO via flotation.

Flotation recoveries achieved from ACO were 83.6% Cu and 80.2% Au. These recoveries include gravity concentration, flotation, and cyanide leaching of flotation tailings.

Cyanide leach testwork conducted to determine the amenability of the ACO and LCO materials to leaching indicates that the preferred processing circuit for LCO is gravity + cyanide leach, and that cyanide leaching of ACO flotation cleaner tailings improves overall ACO Au recovery.

Cyanide leach recoveries achieved from LCO were 92% Au from gravity concentration and cyanide leaching of gravity tails.

Overall testwork shows flotation gold metallurgical recovery of 80% and cyanide leach gold metallurgical recovery of 92%. Further, testwork shows that the preferred processing circuit for LCO is a gravity + cyanide leach while that for ACO is gravity + flotation+ flotation tail leach.

In addition to the primary hardrock ACO/LCO materials, saprolitic cover material was also tested for amenability to gravity concentration, flotation, and cyanide leaching. Gravity and leach recovery testwork indicate that >90% Au recoveries were achieved. Flotation recoveries for the saprolite cleaner test was 80%. Recoveries achieved for 72-hour whole ore cyanide leaching was approximately 98% for both run of mine (RoM) saprolite fines and coarse saprolite ground to 80% passing (P_{80}) of 129 micrometres (µm).

For the Sona Hill deposit, test results indicated that Au from the saprolite composite sample presents high extractions at between 94% and 98% from a flowsheet incorporating gravity+ leach. Au extraction from the Granodiorite Master Composite (GRDT-MC) sample was between 81% to 85% and from the Granodiorite with High Quartz Master Composite (GRDT-QZ) sample, Au was extracted between 74% to 85% using the same flowsheet as the SAP-MC sample. However, increasing the pH of the leach and employing a grind as fine as 53 µm has been found to increase the Sona Hill leach extractions into the high 80% and at times low 90% range for the GRDT-MC and GRDT-QZ composites. The presence of auriferous telluride minerals is believed to be the reason for this leach behaviour. In addition, the use of lead nitrate and pre-aeration assisted in both increased extractions and reduced sodium cyanide consumption.

While the Sona Hill resource does not include Ag as a payable metal, testwork has shown that Ag is present and recoverable in gravity concentration, flotation, and cyanide leaching.

1.5 Mineral Resource Estimate

The Mineral Resource Estimate for the Project conforms to industry best practices and is reported using the 2014 CIM Definition Standard for Mineral Resources and Mineral Reserves and 2019 CIM Best Practice Guidelines. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. This estimate of Mineral Resources may be materially affected by environmental permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

The Mineral Resource Estimate was calculated from two main databases for the Project, one for the Toroparu Deposit and another for the Sona Hill Deposit. Both complete databases are comprised of a total of 709 diamond drill holes and three trenches consisting of 199,996 m. This includes:

- Toroparu Deposit has 528 diamond drill holes consisting of 178,491 m and three trenches comprised of 655.3 m completed between 2006 and 2021, and
- Sona Hill Deposit area has 181 diamond drill holes consisting of 20,850 m completed between 2012 and 2018.

The November 2021 Mineral Resource Estimate was prepared by Nordmin following a two-phase diamond drill program in 2020-2021 which comprised a total of 20,750 m in 114 drill holes. The new drill hole assays were reviewed and fully validated by Nordmin.

Nordmin, through an interactive process with the Company, undertook a full re-examination of the mineralogical, lithological, structural, and geochemical correlations influencing gold (Au) mineralization within the Project. The review concluded that:

- The previous modelling of the mineralization utilizing a single implicit lower grade 0.2 g/t gold shell did not identify nor isolate the structurally controlled higher-grade domains that exist throughout the project area.
- The previous interpretation was not representative of the deposit type nor the geological controls of mineralization that support both lower grade and higher-grade mineralized domains.

- Each domain and corresponding sub domains required extensive modelling of the higher-grade structural domains, which control the higher-grade mineralization within the encapsulating lower grade mineralized domain.

The 2020 and 2021 20,750 m (114 hole) drill program further verified the location and structural relationship between the lower and higher-grade mineralization domains located within the previously defined disseminated lower grade mineralized halo along the 4 km Toroparu trend and for the Sona Hill Deposit. Nordmin incorporated the various geological, structural controls to support the various gold, copper, and silver mineralization styles, and their associated geochemistry. The block model utilized explicit modelling of mineralized structures present in the deposit areas to support the Mineral Resource Estimate. These models incorporate the geologic and structural controls of gold mineralization, the style of mineralization, and its associated geochemistry. The Toroparu Deposit consists of multiple geographical areas, including the Main, NW, and SE Areas. Each of these areas was separated into various domains. The Sona Hill Deposit used three main domains for the estimation process.

The intersection of the NW-SE and E-W structures creates zones of wider and higher-grade gold mineralization than in the structures themselves. These structural intersections occur over a consistent and repeatable pattern that enriches gold, silver, and copper mineralization throughout the deposits. The recognition of these patterns supports the combination of open pit and underground mining methods that form the basis of the Mineral Resource Estimate.

The Mineral Resource was classified in accordance with the 2014 CIM Definition Standards and 2019 CIM Best Practice Guidelines. Mineral Resource classifications or “categories” were assigned to regions of the block model based on the Qualified Persons (QPs) confidence and judgment related to geological understanding, continuity of mineralization in conjunction with data quality, spatial continuity based on variography, estimation pass, data density, and block model representativeness, specifically assay spacing and abundance, kriging variance, and search volume block estimation assignment.

For the Toroparu Deposit, the classification was initially applied from the estimation pass. Blocks populated in pass 1 were classified as Measured, blocks populated in pass 2 were classified as Indicated, and blocks populated in pass 3 were classified as Inferred. Subsequently, the block model was analyzed, and it was determined that classification adjustments were required depending on the drilling density required to support an underground or an open pit resource; blocks in the first, second, and third pass that display a relatively high kriging variance were downgraded to a lower classification. For the Sona Hill Deposit, classification was applied directly from the estimation pass. Blocks populated in pass 1 were classified as Measured, blocks populated in pass 2 were classified as Indicated, and blocks populated in pass 3 were classified as Inferred.

The Mineral Resource Estimate, which is summarized in Table 1-1 and Table 1-2. The updated Mineral Resource Estimate includes an open pit and a maiden underground resource estimate within the Toroparu Main & NW and SE Deposits along with the satellite deposits consisting of the Southeast zone (SE) and the Sona Hill satellite gold deposits.

Table 1-1: Mineral Resource Statement for the Toroparu Project

Deposit	Area	Resource Category	Type	Tonnes ('000s)	Au (g/t)	Au oz ('000s)	Cu (%)	Cu lb ('000s)	Ag (g/t)	Ag oz ('000s)
Toroparu	**Main/NW**	Measured	Open pit	98,070	1.21	3,809	0.110	238,112	1.19	3,743
		Indicated		62,531	1.56	3,133	0.100	137,557	0.91	1,828
Toroparu	**SE**	Measured	Open pit	5,121	1.16	190	0.043	4,826	n/a	n/a
		Indicated		2,403	1.14	88	0.052	2,763	n/a	n/a
Sona Hill	**Sona Hill**	Measured	Open pit	6,958	1.85	413	0.008	1,241	1.07	239
		Indicated		4,180	1.66	223	0.008	700	0.85	115
Toroparu	**Main/NW**	Measured	Underground	727	2.84	66	0.072	1,151	0.47	11
		Indicated		4,978	3.21	514	0.091	9,937	0.41	66
Total Measured				**110,877**	**1.26**	**4,479**	**0.100**	**245,330**	**1.12**	**3,993**
Total Indicated				**74,092**	**1.66**	**3,958**	**0.092**	**150,957**	**0.84**	**2,009**
Total Measured & Indicated				**184,969**	**1.42**	**8,437**	**0.097**	**396,286**	**1.01**	**6,002**
Toroparu	**Main/NW**	Inferred	Open Pit	4,018	1.58	204	0.080	7,118	0.66	85
Toroparu	**SE**	Inferred	Open Pit	9	1.67	1	0.040	8	n/a	n/a
Sona Hill	**Sona Hill**	Inferred	Open Pit	1,365	1.28	56	0.006	179	0.54	24
Toroparu	**Main/NW/SE**	Inferred	Underground	8,403	3.53	953	0.091	16,884	0.25	68
Total Inferred				**13,796**	**2.74**	**1,213**	**0.08**	**24,189**	**0.40**	**177**

Table 1-2: Mineral Resource Estimate Summary

	Tonnes ('000s)	Au (g/t)	Au oz ('000s)	Cu (%)	Cu lb ('000s)	Ag (g/t)	Ag oz ('000s)
Open Pit							
Measured and Indicated	179,264	1.36	7,857	0.097	385,198	1.03	5,924
Inferred	5,393	1.50	260	0.061	7,305	0.63	109
Underground							
Measured and Indicated	5,705	3.16	580	0.088	11,088	0.42	77
Inferred	8,403	3.53	953	0.091	16,884	0.25	68
Total							
Measured and Indicated	184,969	1.42	8,437	0.097	396,286	1.01	6,002
Inferred	13,796	2.74	1,213	0.080	24,189	0.40	177

Mineral Resource Estimate Notes

1. Combined Open Pit and Underground Mineral Resources were prepared in accordance with NI 43-101 and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. This estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
2. Underground and Open Pit Mineral Resources are based on a gold price of $1,630/oz. This gold price is the three-year tailing average as of September 30, 2021.
3. Open Pit Mineral Resources comprise the material contained within various Lerchs-Grossmann pit shells at various revenue factors. These revenue factors are as follows: Main/Southeast/NW Zone @ 0.75 revenue factor and Sona Hill @ 1.00 revenue factor. The gold cut-off applied to Open Pit Mineral Resources within the selected pit shells was 0.40 g/t.
4. Underground Mineral Resources comprise all material found within Mineable Shape Optimizer (MSO) wireframes generated at a cut-off of 1.8 g/t gold including material below cut-off.
5. Silver values are not reported for the SE Open Pit Ag contained metal values reported will not equal A tonnes X grade conversion calculation.
6. Assays were variably capped on a wireframe-by-wireframe basis.
7. Specific Gravity was applied using weighted averages to each individual lithology type.
8. Mineral Resource effective date November 1, 2021.
9. All figures are rounded to reflect the relative accuracy of the estimates and totals may not add correctly.
10. Excludes unclassified mineralization located within mined out areas.
11. Reported from within a mineralization envelope accounting for mineral continuity.

Areas of uncertainty that may materially impact the Mineral Resource Estimate include:

- Changes to long term metal price assumptions.
- Changes to the input values for mining, processing, and General & Administrative (G&A) costs to constrain the estimate.
- Changes to local interpretations of mineralization geometry and continuity of mineralized zones.
- Changes to the density values applied to the mineralized zones.
- Changes to metallurgical recovery assumptions.
- Changes in assumptions of marketability of the final product.
- Variations in geotechnical, hydrogeological, and mining assumptions.
- Changes to assumptions with an existing agreement or new agreements.
- Changes to environmental, permitting, and social license assumptions.
- Logistics of securing and moving adequate services, labour, and supplies could be affected by epidemics, pandemics, and other public health crises, including COVID-19, or similar such viruses.

1.6 Mining Methods

1.6.1 Mineral Resources within the PEA Mine Plan

The estimate of mineral resources within the PEA mine plan is effective as of December 1, 2021 and is presented in Table 1-3. The PEA models an open pit and an underground mine with mineral resources within the PEA mine plan containing 6.156 Moz of Au, 3.993 Moz of Ag and 240.2 Mlb of Cu (109.0 kt).

Measured, Indicated and Inferred resources were used for conversion to mineral resources within the PEA mine plan for the open pit and underground designs. The open pit mineral resources within the PEA mine plan are contained within the Toroparu and NW Pits (Toroparu Pit), Sona Hill Pit and SE Pit and are associated with 558 Mt of waste and a LoM stripping ratio of 5.99:1. The underground mineral resources within the PEA mine plan are contained below the Toroparu Pit.

The mineral resources within the PEA mine plan are valid at the time of estimation and include cut-off grade (CoG) assumptions made before the final PEA cash flow model was completed. SRK and Nordmin confirmed the overall project economics are favorable at the approximate four-year moving average Au price of US$1,500/oz Au, an average Ag price of US$20/oz Ag, and an average Cu price of US$3.13/lb Cu.

Table 1-3: Mineral Resources within the PEA Mine Plan

MINERAL RESOURCES WITHIN THE PEA MINE PLAN								
Area	**Resource Category**	**Tonnes ('000s)**	**Au g/t**	**Ag g/t**	**Cu %**	**Contained Au Toz ('000s)**	**Contained Ag Toz ('000s)**	**Contained Cu Tonnes ('000s)**
All Open Pits	Measured	60,117	1.41	1.36	0.11	2,728	2,633	64.6
	Indicated	31,407	1.74	1.12	0.09	1,756	1,126	29.8
	Measured & Indicated	91,525	1.53	1.28	0.10	4,499	3,769	94.5
	Inferred	1,593	1.62	0.89	0.07	83	45	1.1
	All Open Pits Subtotal	**93,118**	**1.53**	**1.27**	**0.10**	**4,567**	**3,804**	**95.5**
Underground	Measured	839	2.73	0.63	0.07	74	17	0.6
	Indicated	5,899	3.24	0.49	0.11	614	92	6.2
	Measured & Indicated	6,738	3.17	0.51	0.10	687	110	6.8
	Inferred	7,447	3.77	0.33	0.09	902	80	6.6
	Underground Subtotal	**14,185**	**3.48**	**0.41**	**0.09**	**1,589**	**189**	**13.4**
All Open Pits & Underground	Measured	60,956	1.43	1.35	0.11	2,802	2,650	65.3
	Indicated	37,306	1.98	1.02	0.10	2,369	1,219	36.0
	Measured & Indicated	98,262	1.64	1.23	0.10	5,187	3,878	101.3
	Inferred	9,040	3.39	0.43	0.09	985	125	7.7
	Grand Total	**107,302**	**1.78**	**1.16**	**0.10**	**6,156**	**3,993**	**109.0**

Source: SRK, 2021 & Nordmin, 2021

<u>Open pit mineral resources within the PEA mine plan notes</u>

- Open Pit mineral resources within the PEA mine plan:
  - The open pit mineral resources within the PEA mine plan are based on a block by block net smelter return calculation based on an Au price of US$1,500/oz, Ag price of US$20.00/oz and Cu price of US$3.13/lb. The PEA cash flow base case used an Au price of US$1,500/oz., Ag price of US$20.20/oz and Cu price of US$3.13/lb;
  - The open pit mineral resources within the PEA mine plan assume complete mine recovery;
  - The open pit mineral resources within the PEA mine plan are diluted at approximately 15-30% (further to dilution inherent in the resource model and assumes selective mining unit of 5 m x 5 m x 5 m for Main and NW Pits and 2.5 m x 2.5 m x 5m for Sona Hill and SE pits);
  - Contained in situ gold ounces do not include metallurgical ACO recoveries of 83.6% Cu and 80.2% Au and gold LCO recoveries of 92.2%;
  - Waste tonnes within the open pit is 558 Mt at a strip ratio of 5.99:1 (waste to ore);

- Costs assumptions are: Mining Costs = US$2.30/t moved, Processing/Tailings Costs = US$15.50/t processed, G&A Costs = $5.95/t processed;
- An open pit CoG of 0.5 g/t-Au saprolite and 0.5 g/t-Au fresh rock was applied to open pit resources constrained by the ultimate pit design; and
- The open pit mineral resources within the PEA mine plan for the Project was calculated by Fernando P. Rodrigues, BSc, MBA MMSAQP #01405QP of SRK Consulting, Inc. in accordance with the Canadian Securities Administrators National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and generally accepted Canadian Institute of Mining, Metallurgical and Petroleum "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines ("CIM Guidelines").

- Underground mineral resources within the PEA mine plan:
    - The underground mineral resources within the PEA mine plan were prepared by B. Wissent, BEng of Nordmin Engineering Ltd., in accordance with NI 43-101 and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. This estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues;
    - The underground mineral resources within the PEA mine plan are based on selected MSO wireframes generated at Au cut-off of 2.0 g/t based on an Au price of US$1,500/oz. A small amount of the underground mineral resources within the PEA mine plan is based on material from development with a marginal Au diluted cut-off of 1.25 g/t. The PEA cash flow base case used an Au price of US$1,500/oz., Ag price of US$20.20/oz and Cu price of US$3.13/lb;
    - The underground mineral resources within the PEA mine plan assumes mining recovery at approximately 80-92.5% for longhole open stoping (LHOS) and 100% for development;
    - The underground mineral resources within the PEA mine plan are diluted at approximately 12% for LHOS and 5% for development; and
    - Costs assumptions are: Mining Costs = US$36.00/t processed, Processing/Tailings Costs = US$15.50/t processed, G&A Costs = US$6.00/t processed, Operating Cost Marginal Allowance (10%) = US$5.80/t processed.
- The mineral resources within the PEA mine plan tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding;
- "g/t" = gram per metric tonne, "Toz" = troy ounces; and
- The mineral resources within the PEA mine plan effective date: December 1, 2021.

1.6.2 Open Pit Mining

A conventional truck-shovel method was considered for the open pit portion of the Toroparu Deposit, as shown in Figure 1-1 and Figure 1-2. The open pit analysis results in several distinct open pits coalescing into the NW and Main Toroparu Pits over time. The Sona Hill and Southeast Zone (SE) will be developed in a similar fashion beginning in year 3 and 6 respectively. The final dimensions of the NW Pit are approximately 990 m long x 690 m wide x 360 m deep. The dimensions of the Main Pit are approximately 1,300 m long x 750 m wide x 470 m deep. The open pit LoM plan proposes to mine approximately 93 Mt at a cut-off grade of 0.5 g/t Au and 558 Mt of waste rock material. The average stripping ratio for the open pit operations is 6:1 over the LoM. Each pit is currently planned to be developed with 29 phases each. Compacted saprolitic waste material will be used to construct haul roads, facility pads and flood control berms, levies, and other structures.

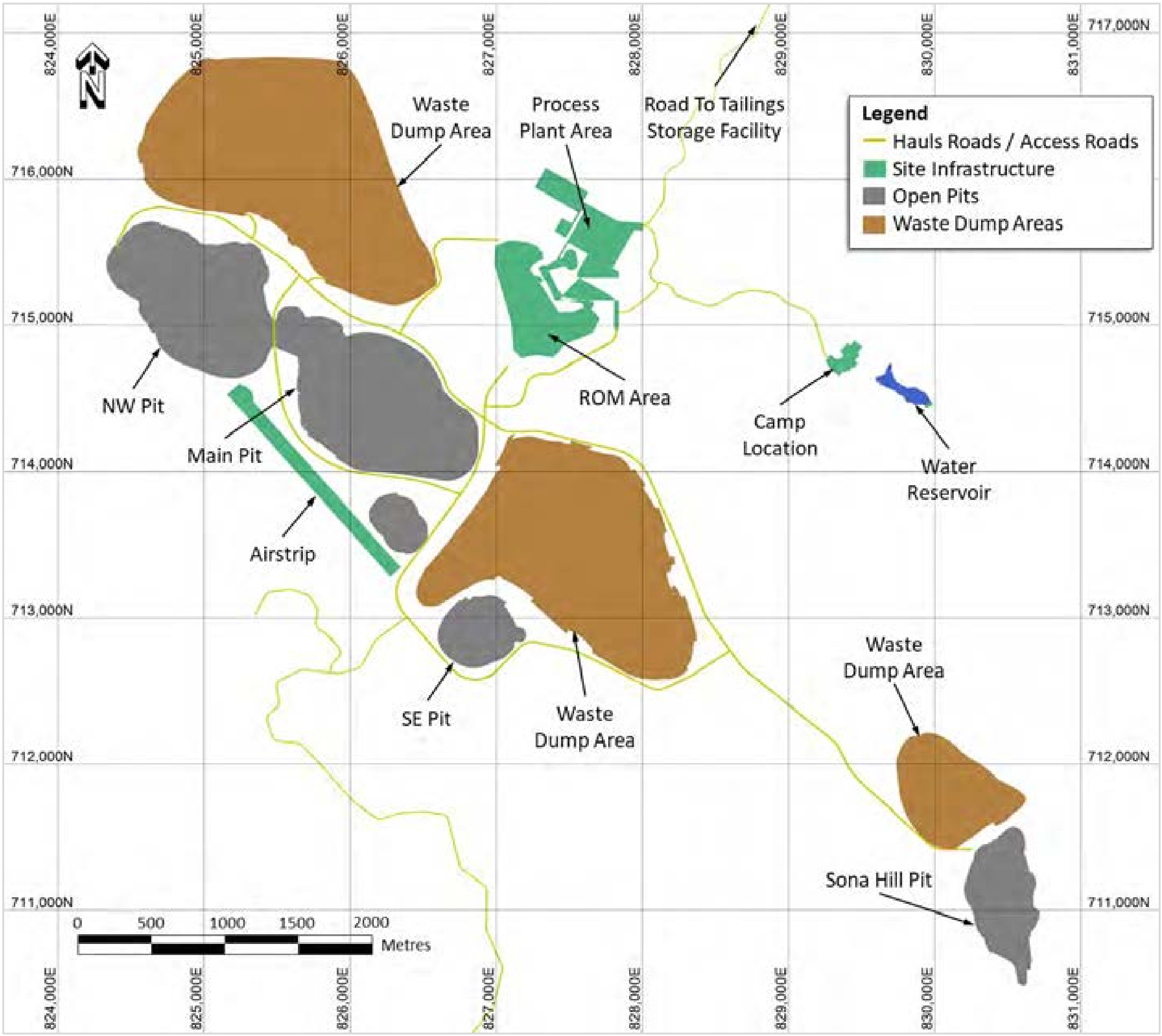


Figure 1-1: Open pits, waste dumps and site infrastructure, plan view

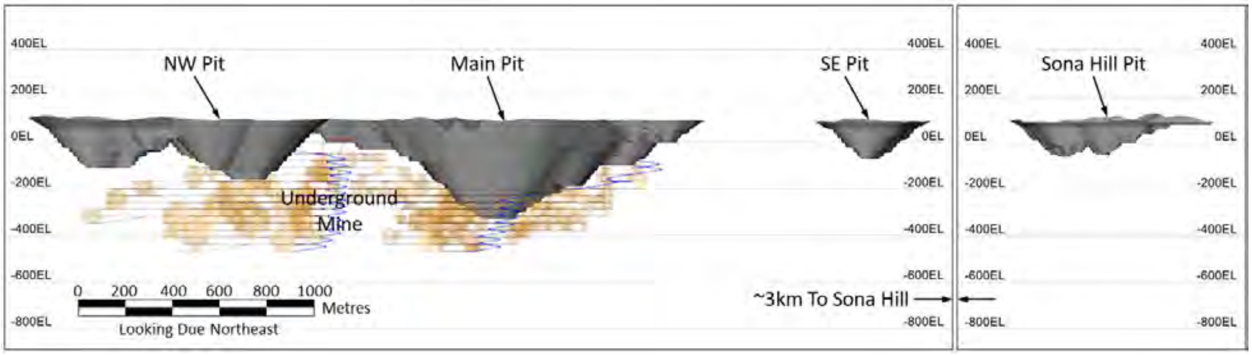


Figure 1-2: Open pits, long section view (looking Northeast)

1.6.3 Underground Mining

Underground development will commence at the beginning of the ninth year of open pit operation and targets 3,500 tpd, ramping up to full production over an approximately two-year period. The ramp-up allows for the main ramp system development at the 250 m elevation down from the surface portal in the NW Pit and to connect to both fresh air and return air raises, providing ventilation and secondary egress for the mine. Underground production is scheduled based on approximately 3,500 tpd mill feed and 750 tpd average waste, excavated using a fleet of 15 and 10 tonne load-haul-dump loaders, hauled with 45 tonne trucks using the ramps to portals entrances and rehandled using the surface fleet. Production is expected to commence in the central area between the Main and NW Pits from 360 Level (approximately 360 m elevation below surface) and continues for the first 2 years in a bottom-up sequence. It is anticipated that mining next transitions to production from lower mining areas below and around Main and NW Pits for approximately the final 10 years of the LoM. Figure 1-3 shows the LoM underground design.

The underground mineralization was evaluated using Datamine's MSO tool to create the mineable inventory. The mining cut-off for the MSO underground inventory was generated based on a 2.0 g/t gold cut-off grade (insitu), which approximately equates to a 1.6 g/t gold mill feed grade. A 1.25 g/t incremental mill feed cut-off grade was selected to apply to development. Stopes were created on 30 m level spacing and a maximum of 15 m length, with an average mineralized width of approximately 7 m. Stopes are mined via longitudinal retreat and are accessed by overcut and undercut stope access drifts which extend from the level haulages. The LoM underground mill feed is approximately 14.18 Mt at an average gold grade of 3.48 g/t, and 3.49 Mt of waste.

This PEA is preliminary in nature. In addition to the Measured and Indicated Resources, the mine plan presented in this section includes Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that this PEA will be realized.

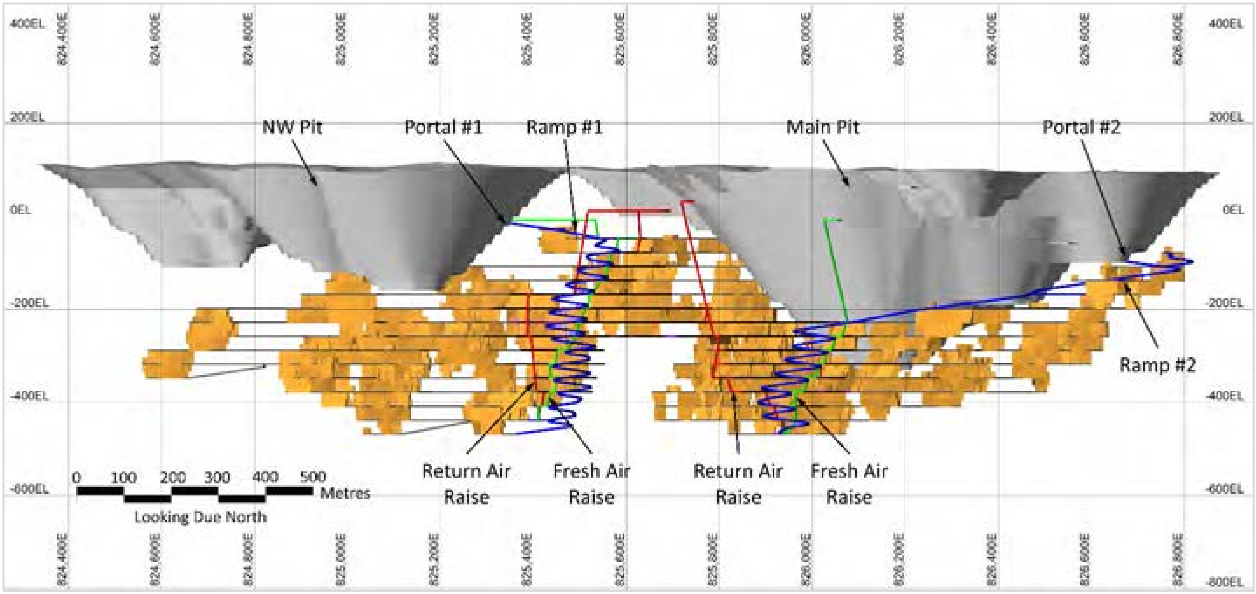


Figure 1-3: Underground long section view (looking northeast)

1.7 Recovery Methods

The concentrator is designed to process 14,000 tonnes per day (tpd) of mineralized material (nominal) during its peak operation. The processing plant will be constructed in two phases. The first phase consists of the initial 5 years of the Project where the plant will receive LCO and saprolitic material to recover Au at a nominal throughput of 7,000 tpd. During the second phase, the plant will be expanded with the addition of a 7,000 tpd Cu flotation circuit and the associated equipment to process ACO and produce a Cu concentrate. The overall plant capacity will double in size to 14,000 tpd with the addition of the flotation circuit.

Phase 1 processes 7,000 tpd of LCO and saprolite material through crushing and grinding, carbon in leach (CIL) circuit and adsorption, desorption and recovery (ADR) to produce Au doré. This phase continues through the LoM. Sona Hill material will also be processed during this period.

In Phase 2, ACO will be processed at 7,000 tpd of ACO through flotation with cyanide leaching of the rougher scavenger concentrate and cleaner flotation tailings via a CIL circuit. Based on metallurgical testwork recovery by flotation, a Cu concentrate with grade of approximately 21% Cu is expected to be produced.

Gravity concentration with intense cyanidation will be performed on a portion of the underflow from the grinding cyclones in the Gold Plant and similarly in the Flotation Plant once it comes on line.

1.8 Project Infrastructure

The Project is a greenfield gold project that will have supporting infrastructure both on and off site. Existing facilities on site including an exploration camp, airstrip, and site roads.

1.8.1 On Site Infrastructure

The on site facilities will include a security entrance, site access roads, mine haul roads, open pit mine and waste rock storage areas, processing plant, laboratory and associated shops and offices, fuel storage and delivery facility, fuel oil generating facility, explosives storage facility, camp, administrative buildings, emergency treatment facility, shops, warehouses, an airstrip, and laydown yards. The on site project facilities will be supported by services and utilities.

The utilities and services will include potable water systems, water supply system, and firewater system. An on site landfill will be utilized. The site will include a sewage collection, treatment, and disposal system. Additionally, a full communications system including radio, satellite, and a regional mobile telephone tower and system will be constructed. A fibre optic network will be installed throughout the site.

1.8.2 Site Water Management Facilities

The purpose of site water management structures (WMS) include:

Manage the Wynamu River and protect mine facilities for events up to the 100-year 24-hour storm;

- Develop a wetland within the Wynamu River Drainage ("Wetland") to retain all site water for de-sedimentation prior to release to the environment via the Puruni River (Figure 18-3);
- Divert non-contact water to the Wetland;
- Collect contact water in ditches and convey it to the Wetland; and
- Release water from the Wetland to the Puruni River.

The WMS include mine haul roads acting as WMS structures, contact and non-contact diversion ditches, the Wetland and culverts regulating the flow and impoundment of site water for sedimentation.

1.8.3 Tailings Storage Facility

The TSF, located on the northeast side of the mine property, will be organized and operated in the area encompassing Module 1 and Module 3 (developed by KCB (2014)). The TSF has been designed for a storage capacity of 107 Mt within an impoundment with a final nominal capacity of 156 Mt of slurry tailings.

Tailings will be confined by site topography and constructed saddle dams, with the typical section being compacted saprolite shells with a chimney drain to relieve the head from the pond in the centre of the final dam and conduct seepage through finger drains downstream.

Water balance estimates indicate excess water volumes (mainly due to precipitation) during operations of the tailings modules. Water management includes the use of diversion channels and discharge of excess water volumes to the environment through operating spillways designed for the Probable Maximum Flood (PMF).

1.8.4 Off Site Infrastructure

The off site facilities will include port access and access to the Project by road. The port facilities are located near Itaballi at a location on the south bank of the Cuyuni River approximately 2 miles upstream of the confluence with the Mazaruni River known as Pine Tree.

1.8.5 Project Logistics

During construction and mine operations, transportation of equipment, materials, and supplies will be delivered by barge and truck from Georgetown Harbor to a newly constructed port/wharf at Pine Tree and overland to the Project, and by air from Ogle International Airport in Georgetown.

Logistical infrastructure includes docking and transshipment facilities at third party ports in Georgetown, docking/unloading facilities for barges at the Pine Tree Port facility near Itiballi, overland access from Pine Tree to the Toroparu South Junction on the Itiballi-Puruni Landing-Papishao public road, then private road from the Toroparu South Junction to the Project site.

1.9 Environmental Studies, Permitting and Social Impact

The Property is located within the Mazaruni Mining District, one of six mining districts in Guyana. This mining district is located within Region 7 of Guyana, the Cuyuni–Mazaruni Region, one of ten administrative regions within the Country of Guyana.

The Property is held and operated through ETK Inc. (ETK), the Company's wholly owned subsidiary.

ETK holds the mineral properties in the Upper Puruni Area. They are comprised of seven Small Scale claims, 65 Prospecting Permits Medium Scale (PPMS), 25 Mining Permits (MP) and two contiguous Prospecting Licenses (PL) that collectively cover an area of 105,802 acres or 42,816.55 ha.

1.9.1 Environmental Studies

The initial environmental baseline studies were conducted in 2007, 2008 and 2010. The results were summarized, and the impacts were interpreted as part of the Environmental Impact Assessment (EIA) submittal (ETK, 2012). Subsequent environmental studies included geochemical characterization. Baseline data on the physical environment and biodiversity were recorded and observed over an initial baseline period of May 2007 to May 2008 and supplemented with additional information collected in June – July of 2010. An initial baseline study was done for the Sona Hill area in 2018. Addendums to these baseline investigations will be necessary for the expansion areas of the proposed PEA mine plan, including, but not necessarily limited to the Sona Hill Pit and waste rock dump areas. It is not anticipated at this time that these areas will differ materially from those areas already studied. Summaries of the existing baseline data programs are included in Section 20, and include:

- Surficial Soils;
- Climate;
- Air Quality;
- Surface Water;
- Groundwater;
- Archaeological Resources;
- Flora; and
- Fauna (Terrestrial, Avifauna, Herpetofauna, and Special Interest Species).

Geochemical characterization studies were conducted by Klohn Crippen Berger (KCB) from 2011 to 2013 on the dominant bedrock lithologies representing waste rock and low-grade economic material, and metallurgical tailings representing the three main economic material types. Results of the solid-phase elemental analysis indicated that the lithologies included high concentrations of silver, arsenic, cobalt, chromium, copper, nickel, molybdenum, sulphur, and selenium in comparison to average crustal abundance of high-calcium granite. There was a wide variation between the different lithologic units. The paste pH results indicated that the major lithologies are alkaline with the exception of the saprolite and the Transition Zone samples. The saprolite samples were slightly acidic to neutral while the transition zone samples were neutral to alkaline. These results indicate that no acidity was released from any of the samples except from the saprolite samples. The alkaline results indicate effective carbonate buffering. The net acid generation (NAG) pH results confirmed the not-potentially acid generating (NPAG) acid rock drainage (ARD) risk of waste rock and low-grade economic material samples. Humidity cell testing (HCT) was recommended to be completed to further assess metal leaching of waste rock, low-grade economic material and open pit walls under alkaline conditions. The tailings samples were classified as NPAG based on the sulphide-sulphur values and are, therefore, considered to have negligible risk of ARD (KCB, 2013).

Although additional studies are recommended to further develop mining waste management strategies and characterize the PEA-proposed expansion areas, there do not appear to be any known environmental issues that could materially impact the Company's ability to extract the mineral resources at the site. Preliminary mitigation strategies have been developed to reduce environmental impacts to meet regulatory requirements and the specifications of the environmental permit.

The overall environmental management objective of the Project is to use best available techniques (BATs), best management practices (BMPs) and modern, proven technology to operate a gold and copper mine, process plant, and supporting infrastructure consistent with the social, economic and environmental requirements of the Government of Guyana and, to the extent that they represent recognized international BMPs and World Bank/IFC/Equator Principles policies and guidelines. The Company will establish and maintain a documented, comprehensive Environmental and Social Management System (ESMS) over the construction, operation and closure phases of the Project.

1.9.2 Project Permitting

The Mining Act of 1989 governs the establishment of a mine and appoints the Guyana Geology and Mines Commission (GGMC) as the state agency with responsibility for mining in Guyana. In addition to the Mining Act; the Amerindian Act, the Environmental Protection Act, and the Occupational Health and Safety Act also set out conditions relevant to the development of a mine.

For large scale operations, the operator is required to apply for a mining license. The process for the application of a mining license requires that the applicant submit a technical and economic feasibility study, processing and mine plans, and an EIA. A mining license is valid for 20 years, or for the life of the mine if it is shorter and can be renewed at the end of the first 20 years, if needed. A mining license is only granted after all the prerequisite conditions have been met. The license holder must pay an annual rental fee for each acre within the mining permit. The rate for a mining permit is set out by the GGMC and updated periodically. In some cases, a performance reclamation bond may be required.

The applicable permit or license requirements, and the status of any permit applications, are presented in Section 20.

The Project received environmental permits for gold and copper mining and processing in 2012 based on an original permit application dated May 2, 2008, and the approved EIA (ETK, 2012). The permit was issued to ETK. The permit included design, operational and reporting compliance items.

ETK submitted an amendment to its Environmental Management Plan in October of 2021 to include the processing of silver from the deposits and adding the Southeast Area of the Main Toroparu Deposit and the Sona Hill Deposit to the permitted operations under the Environmental Authorization. EPA accepted the revised Environmental Plan on November 22, 2021.

1.9.3 Social and Community

The socio-economic and socio-cultural baseline was compiled based on literature review and on field surveys conducted in communities considered to be within the Project area of influence. The study details and interpretation were presented in the EIA (ETK, 2012).

1.9.4 Mine Closure

The license holder is responsible for mine closure and reclamation. In addition to the EIA and permit closure discussions, KCB completed the Toroparu Project Conceptual Mine Reclamation and Closure Plan in 2017. IFC and other international standards, regulations and baseline information was considered for the closure plan, to achieve the following objectives:

- Prevent, reduce or mitigate the long term adverse environmental and social effects associated with the Project;
- Reduce the need for long term monitoring and maintenance by designing for closure using current available proven technologies and instituting progressive reclamation;
- Provide mine landscapes that are in a geotechnically and geochemically stable and safe condition;
- Provide for the return of all affected ecosystems to healthy and sustainable functioning; and
- Provide for long term monitoring and maintenance affected by the Project.

Performance standards to measure closure success (assumed to be achieved within 20-years post-closure) are as follows:

- Physical stability (static) to a factor of safety of 1.5 for remaining facilities;
- Biological stability on 70% of site areas intended for revegetation;
- Chemical stability of mine wastes to prevent water degradation and impacts to humans or wildlife; and
- Water quality similar to or improved when compared to background pre-mining baseline data.

The PEA anticipates cessation of milling and processing in Year 24, with a closure cost expenditure occurring entirely in Year 25. The base allowance for closure costs presented in the Technical Economic Model is US$22,216,000 with an allowance for a 30% contingency making the total closure cost estimate for the Project to be US$28,881,000. Nordmin did not prepare this estimate, nor were the calculations provided for Nordmin's review. However, the estimate is consistent with the reclamation cost estimate attached to the most recent closure plan (KCB, 2017), and is in keeping with other gold mining operations of similar size.

No post-performance or reclamation bond was specified in the approved EIA issued by the Guyana EPA; however, a detailed closure plan is required two years prior to scheduled closure and the plan will be subject to agency approval. A bond may be specified and required as part of the modification process for the proposed PEA operation and amended EIA.

1.10 Capex and Opex Costs

The total estimated initial cost to design and construct the Project identified in this report is US$355 million. Approximately US$41 million of this estimate is related to pre-stripping costs and the remainder of US$314 million is directly related to the installation of the Project site facilities and purchasing of equipment.

Initial capital will support the installation of a leaching circuit that will produce doré bars bearing gold and silver and will operate at a feed rate of 7,000 tpd, this circuit will support the operation for the first five years.

In years four and five expansion capital will be used to install a flotation circuit that will operate at a feed rate of 7,000 tpd, bringing the total project feed rate to 14,000 tpd, and will produce a copper concentrate bearing copper, gold and silver. This circuit will begin operating in year 6 and its cost is estimated at US$281 million (including expansion of the mine fleet, processing circuit, infrastructure, power and associated indirect and owner's costs). The free cash flow from the Project is estimated to self-finance this expansion.

This PEA's capital cost estimates consider the precious metal purchase agreement (PMPA) with Wheaton Precious Metals Corp. (Wheaton) for the purchase of 10% of the gold produced over the LoM at US$400 per payable ounce; and 50% of the silver produced over the LoM at US$3.90 per payable ounce. The acquisition cost of this precious metal production stream of US$138 million and is entirely included as a payment towards the initial capital. This acquisition cost is used to reduce the Project's capital requirements in the economic model and are identified in this report as PMPA Installments.

Sustaining capital is estimated at US$662 million for the LoM and will support equipment maintenance and replacement, incremental capacity increases, water management structures and tailings storage facility expansions, infrastructure maintenance and associated indirect and owner's costs.

The aggregate capital estimate is considered to be within a ±40% weighted average accuracy of actual costs. Base pricing will be in Q4 2021 US dollars, with no allowances for inflation or escalation beyond that time.

The contingency cost is based on the following factors of specific direct cost areas:

- Leaching Process Circuit: 28.60%
- Flotation Process Circuit: 26.23%
- Off site Infrastructure: 5.00%
- On site Infrastructure: 10.00%
- Water Management and Treatment: 15.00%
- Tailings Storage Facility: 15.00%
- Buildings and Ancillary Equipment: 20.00%
- Closure: 30.00%

The total contingency represents roughly 17% of the direct cost estimates from the initial and expansion capital. The contingency is included to account for unanticipated costs within the scope of the estimate. The percentage allowances were individually assessed based on the accuracy of the quantity measurement, type and scope of work, and price information for the capital cost estimate.

The estimate is based on first principles estimates based on vendor quotations and cost databases from similar projects. It does not reflect discounts for negotiated prices, bulk purchasing, or used equipment purchases where appropriate, any of which could lead to reductions in actual capital costs relative to the prices used in the capital estimate.

A summary overview of the estimate by area is presented in Table 1-4.

Table 1-4: Summary of Capital Costs by Area

PEA Capital Cost Estimates (US$M)	Scope	Initial Capital (Pre-Prod) (US$M)	Expansion (US$M)	Sustaining Capital (US$M)	LoM Capital (US$M)
Mine (Open Pit & Underground)	SRK/Nordmin	24	69	601	695
Process Plant	Metifex	95	103	-	198
Water and Tailings Management	KCB	17	-	29	45
Infrastructure	GCM Mining	64	-	6	69
Power Supply	GCM Mining	3	-	-	3
Owner's	GCM Mining	23	21	22	66
Indirect Costs	GCM Mining	52	61	-	113
Risk and Contingency	GCM Mining	36	27	4	67

PEA Capital Cost Estimates (US$M)	Scope	Initial Capital (Pre-Prod) (US$M)	Expansion (US$M)	Sustaining Capital (US$M)	LoM Capital (US$M)
Subtotal Capital Expenditures		**314**	**281**	**662**	**1,258**
Capitalized Rock Pre-Stripping	SRK	41	-	-	41
PMPA Installments	GCM Mining	(138)	-	-	(138)
Net Financing Required		217	281*	662*	1,161

Source: SRK/Nordmin/Metifex/KCB/GCM Mining, 2021. * Free Cash Flow is sufficient to finance

Capital costs exclude:

- Escalation;
- Taxes (Value Added Tax [VAT]); and
- Import Duties.

Imported equipment, materials, and operating supplies are not subject to taxes (VAT), import or other duties as per the Mineral Agreement with the government of Guyana.

The operating cost estimates have been assembled by area and component, based upon estimated staffing levels, consumables, and expenditures according to the mine and process design. LoM operating costs are shown in

Table 1-5, and annual operating costs in Table 1-6 (rounded to nearest US$1,000,000).

Table 1-5: Operating Cost LoM

Area	Expenses (US$M)	US$/t Mined	US$/t-Mill
Mine	1,841	2.65	17.16
Processing	1,558	n/a	14.52
G&A	360	n/a	3.36
Total Operating	**3,758**	**n/a**	**35.03**

Source: SRK, 2021.

Table 1-6: Annual Operating Cost, US$ x 1,000,000

Period	Ore Milled (Mt)	Mining (US$M)	Processing (US$M)	G&A (US$M)	Total (US$M)	US$/t milled
-3	-	-	-	-	-	-
-2	-	-	-	-	0	-
-1	-	-	-	-	0	-
1	2.18	(42)	(35)	(14)	(91)	(41.99)
2	2.56	(43)	(39)	(14)	(96)	(37.45)
3	2.56	(42)	(38)	(15)	(95)	(37.19)
4	2.56	(45)	(39)	(15)	(99)	(38.67)
5	2.56	(44)	(40)	(15)	(99)	(38.79)
6	4.73	(41)	(69)	(16)	(126)	(26.71)
7	5.11	(45)	(74)	(17)	(135)	(26.50)

8	5.11	(54)	(74)	(17)	(144)	(28.17)
9	5.12	(55)	(74)	(17)	(145)	(28.28)
10	5.11	(56)	(73)	(17)	(146)	(28.63)
11	5.11	(96)	(73)	(17)	(186)	(36.36)
12	5.11	(116)	(73)	(16)	(205)	(40.18)
13	5.12	(114)	(74)	(16)	(204)	(39.75)
14	5.11	(120)	(73)	(16)	(209)	(40.97)
15	5.11	(119)	(73)	(14)	(207)	(40.46)
16	5.11	(115)	(73)	(14)	(202)	(39.63)
17	5.12	(117)	(73)	(14)	(205)	(40.00)
18	5.11	(113)	(73)	(14)	(201)	(39.28)
19	5.11	(112)	(73)	(14)	(199)	(39.04)
20	5.11	(108)	(73)	(14)	(195)	(38.19)
21	5.12	(110)	(73)	(14)	(197)	(38.44)
22	5.11	(85)	(74)	(13)	(173)	(33.86)
23	5.11	(39)	(74)	(13)	(125)	(24.56)
24	3.25	(9)	(50)	(14)	(72)	(22.25)
25	-	-	-	-	0	-
Total	**107.30**	**(1,841)**	**(1,558)**	**(360)**	**(3,758)**	**(35.03)**

Source: SRK, 2021

1.11 Economic Analysis

Project evaluation resulting economics present an after-tax net present value of US$794 million, at 5% discount rate, and an internal rate of return of 46.08%. Table 1-7 presents further details of the economic evaluation.

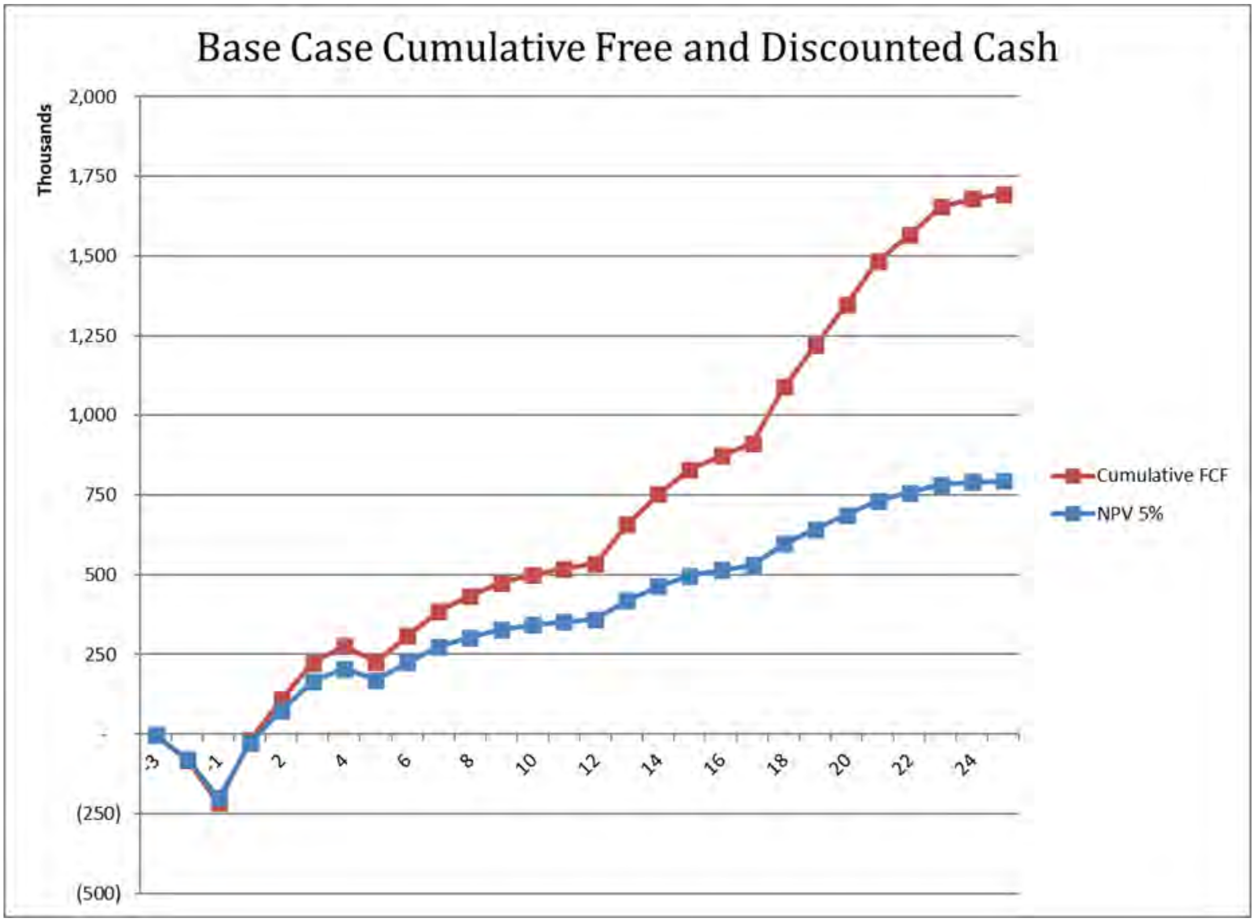
Table 1-7: Project Evaluation Economic Results

Description	Value US$000's
Metal Prices	
Au – Sold to Market (US$/oz)	$1,500
Au – Sold to WPM (US$/oz)	$400.00
Ag – Sold to Market (US$/oz)	$20.22
Ag – Sold to WPM (US$/oz)	$3.90
Copper	$3.13
Estimate Of Cash Flow (All Values in US$000's)	
Gross Income	
Payable Au (Doré+Concentrate)	$7,516,386
Payable Ag (Doré+Concentrate)	$30,413
Payable Copper (US$/lb)	$441,660
Gross Income	**$7,988,460**
Treatment Charges	($18,634)
Refining Charges	($18,019)

Description	Value US$000's
Predicted Penalties	($1,553)
Freight Insurance Cost	($63,508)
Gross Revenue	**$7,886,746**
Guyana Au Royalty	($595,750)
Guyana Ag Royalty	($2,018)
Guyana Cu Royalty	($6,185)
One Time Royalty to Surface Owner	($20,000)
Net Revenue	**$7,262,794**
Operating Costs	
Mining Cost	($1,840,872)
Processing Cost	($1,557,511)
Site G&A Cost	($360,096)
Total Operating	**($3,758,479)**
/t ore	($35.03)
Cash Cost (/Au oz)	($695)
Operating Margin (EBITDA)	**$3,504,315**
Initial Capital	($354,760)
Sustaining Capital	($943,811)
PMPA Installments	$138,000
Income Tax	($649,572)
Free Cash Flow	**$1,694,172**
After-Tax IRR	**46.08%**
After-Tax Present Value 5%	**$794,034**
After-Tax Present Value 8%	**$535,423**
After-Tax Present Value 10%	**$420,676**

Source: SRK, 2021

The base case payback period is estimated at 1.15 years. Figure 1-4 presents the cumulative free and discounted cash flow profile.



Source: SRK, 2021

Figure 1-4: Cumulative free and discounted cash flow

The economic modelling resulted in a LoM cash cost of US$916/Au oz, as presented in Table 1-8.

Table 1-8: Project LoM Cash Cost

PEA Cash Cost Estimates	LoM Average ($/oz. Payable Gold)
Mining Cost (open pit & underground)	$340
Processing Cost	$288
Site G&A Cost	$67
Freight & Insurance Cost	$12
Treatment Charges	$3
Refining Charges	$3
Predicted Penalties	$0
By-Product Credit	($87)
Royalties	**$115**
Cash Cost	**$742**
Sustaining Capital	$175
All-in sustaining cash cost (AISC)	**$916**

Source: SRK, 2021

Table 1-9 shows annual production and cash flow forecasts for the life of the Project.

Table 1-9: Project LoM Cash Cost

Years	Mined Resource (kt)	Leaching Feed (kt)	Flotation Feed (kt)	Payable Au Produced (kozs)	Payable Ag Produced (kozs)	Payable Cu Produced (klbs)	Free Cash US$000's	Discounted Cash US$000's
-3	-	-	-	-	-	-	(3,474)	(3,474)
-2	-	-	-	-	-	-	(78,164)	(74,442)
-1	2,214	-	-	-	-	-	(135,122)	(122,559)
1	4,212	2,178	-	280	70	-	198,092	171,120
2	3,373	2,555	-	228	57	-	125,650	103,373
3	4,684	2,555	-	204	104	-	116,407	91,208
4	3,796	2,555	-	207	63	-	51,782	38,641
5	4,757	2,562	-	189	24	-	(47,837)	(33,997)
6	4,036	2,555	2,172	171	152	8,012	80,820	54,702
7	4,300	2,555	2,555	165	212	9,628	77,413	49,901
8	4,066	2,555	2,555	160	136	7,695	47,569	29,203
9	5,080	2,562	2,562	213	163	8,670	41,343	24,172
10	5,480	2,555	2,555	203	179	9,696	25,152	14,006
11	4,369	2,555	2,555	198	97	10,521	19,362	10,268
12	3,812	2,555	2,555	225	77	7,576	15,170	7,662

Years	Mined Resource (kt)	Leaching Feed (kt)	Flotation Feed (kt)	Payable Au Produced (kozs)	Payable Ag Produced (kozs)	Payable Cu Produced (klbs)	Free Cash US$000's	Discounted Cash US$000's
13	7,199	2,562	2,562	290	155	11,680	123,543	59,427
14	3,065	2,555	2,555	248	124	6,604	94,551	43,315
15	3,541	2,555	2,555	245	74	4,357	78,162	34,102
16	3,787	2,555	2,555	214	83	6,597	42,581	17,693
17	3,111	2,562	2,562	232	58	6,907	40,122	15,878
18	4,942	2,555	2,555	340	91	5,749	175,891	66,292
19	7,227	2,555	2,555	352	124	7,336	130,818	46,956
20	7,763	2,555	2,555	275	140	7,265	127,168	43,472
21	5,832	2,562	2,562	300	115	7,523	137,772	44,854
22	3,767	2,555	2,555	203	98	9,321	80,626	25,000
23	2,523	2,555	2,555	173	56	4,404	88,178	26,039
24	368	1,899	1,346	92	72	1,754	26,155	7,356
25	-	-	-	-	-	-	14,442	3,868
26	-	-	-	-	-	-	-	-
Total	**107,302**	**60,322**	**46,981**	**5,407**	**2,522**	**141,295**	**1,694,172**	**794,034**

Source: SRK, 2021

1.12 Interpretations and Conclusions

The results of the PEA affirm the Project's technical and financial merits using base case and sensitivity metal price assumptions and the inputs in some areas from advanced historical studies completed by the Company that were at PFS or FS levels.

The Company believes there is further potential to significantly expand the Mineral Resource and the mineral resources within the PEA mine. Under the assumptions presented in this Technical Report, and based on the available data, the Mineral Resources meet 2014 CIM Definition Standards, the 2019 CIM Best Practice Guidelines and show reasonable prospects of eventual economic extraction.

1.13 Recommendations

The recommendations are focused on the completion of a PFS technical report that is predicated on additional infill drilling to increase the confidence of the resource, carry out further metallurgical test work, and various mine planning, processing related trade-off studies.

Table 1-10 tabulates the PFS recommendations which are anticipated to require a budget of US$7,826,500.

Table 1-10: Recommended PFS Budget

Item	Units	Unit Cost	Cost (US$)
10,000-metre infill drilling expansion drilling (used for metallurgy, geotechnical, etc.)	15,000	$200	$3,000,000
Underground Reserve development			$200,000
TSF and WMS Geotechnical Investigations			$240,000
Metallurgical and Comminution Testwork			$575,000
Mine Design Updates			$600,000
PFS Level Economic Assessment & Technical Report 43-101 (PFS)			$600,000
General support and administration costs, legal fees, professional fees, staff, fixed costs, etc.		40%	$1,900,000
Contingency (10%)		10%	$711,500
Total			**$7,826,500**

2 INTRODUCTION

2.1 Terms of Reference

This Technical Report was prepared as a NI 43-101 Technical Report and a Preliminary Economic Assessment for GCM Mining by Nordmin for the Project situated in the Upper Puruni River Region of western Guyana, South America.

The Mineral Resources are considered effective as of November 1, 2021. This Technical Report supersedes all prior technical reports, Mineral Resource Estimates, and Preliminary Economic Assessments prepared for the Project. As of the date of this report, the Company anticipates using these Mineral Resources for future drill targeting and Mineral Resource upgrades to Mineral Reserves.

GCM Mining is a TSX and OTCQX listed mid-tier gold producer, with their head office located at:

401 Bay Street, Suite 2400, PO Box 15
Toronto, Ontario M5H 2Y4
Canada

The quality of information, conclusions, and estimate contained herein are consistent with the level of effort involved in Nordmin's services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications outlined in this Technical Report.

This Technical Report is intended to be used by the Company; this permits the Company to file this report on SEDAR as a NI 43-101 Technical Report with the Canadian Securities Administrators. Nordmin understands that the Company may use this Technical Report for a variety of corporate purposes. The responsibility for this disclosure remains with the Company. The user of this document should ensure that this is the most recent Technical Report for the Toroparu Gold Project, as it is not valid if a new Technical Report has been issued.

This Technical Report provides a Mineral Resource and classification of the Mineral Resource prepared in accordance with the CIM, Metallurgy and Petroleum Standards on Mineral Resources and Reserves: Definitions and Guidelines, May 10, 2014 (CIM, 2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (November 2019; 2019 CIM Best Practice Guidelines).

2.2 Qualified Persons

The consultants preparing this Technical Report are specialists in the fields of geology, exploration, mineral processing, metallurgical testing, infrastructure and logistics, tailing storage, site water management, open pit and underground mine design and Mineral Resource, and Mineral Reserve estimation, and classification.

Nordmin nor any associates employed in the preparation of this Technical Report are insiders, associates, affiliates, or has any beneficial interest in the Company. The results of this Technical Report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between the Company and Nordmin. Nordmin is being paid a fee for the work in accordance with reasonable professional consulting practices.

This Technical Report was prepared by the QPs listed in Table 2-1, and their responsibilities for each section are indicated. By virtue of their education, experience, and professional association, these individuals are considered a QP as defined in the NI 43-101 Instrument and are members in good standing of a relevant professional institution. The QP Certificates of the Authors are provided in Appendix A of this Technical Report.

Table 2-1: QP – Section Responsibility

Section and Title	Qualified Person	Company
1: Summary	Various QPs	Various
2: Introduction	Glen Kuntz, P.Geo	Nordmin
3: Reliance on Other Experts	Glen Kuntz, P.Geo	Nordmin
4: Property Description and Location	Glen Kuntz, P.Geo	Nordmin
5: Accessibility, Climate, Local Resources, Infrastructure, and Physiography	David Willms, P.Eng., Glen Kuntz, P.Geo	KCB, Nordmin
6: History	Glen Kuntz, P.Geo	Nordmin
7: Geological Setting and Mineralization	Glen Kuntz, P.Geo	Nordmin
8: Deposit Types	Glen Kuntz, P.Geo	Nordmin
9: Exploration	Glen Kuntz, P.Geo	Nordmin
10: Drilling	Glen Kuntz, P.Geo	Nordmin
11: Sample Preparation, Analyses, and Security	Glen Kuntz, P.Geo	Nordmin
12: Data Verification	Glen Kuntz, P.Geo	Nordmin
13: Mineral Processing and Metallurgical Testing	Kurt Boyko, P.Eng.	Nordmin
14: Mineral Resource Estimate	Glen Kuntz, P.Geo	Nordmin
15: Mineral Reserve Estimate	Fernando Rodrigues, MMSAQP	SRK
16: Mining Methods	Fernando Rodrigues, MMSAQP, Brian Wissent, P.Eng., Daniel Yang, P. Eng. and Ben Peacock, P. Eng.	SRK, Nordmin & Knight Piésold
17: Recovery Methods	Kurt Boyko, P.Eng.	Nordmin
18: Project Infrastructure	David Willms, P.Eng., Kurt Boyko, P. Eng.	KCB, Nordmin
19: Market Studies and Contracts	Glen Kuntz, P.Geo	Nordmin
20: Environmental Studies, Permitting, and Social, or Community Impact	Glen Kuntz, P.Geo	Nordmin
21: Capital and Operating Costs	Fernando Rodrigues, MMSAQP, Kurt Boyko, P.Eng., Brian Wissent, P.Eng.	SRK, Nordmin
22: Economic Analysis	Fernando Rodrigues, MMSAQP	SRK
23: Adjacent Properties	Glen Kuntz, P.Geo	Nordmin
24: Other Relevant Data and Information	Glen Kuntz, P.Geo	Nordmin
25: Interpretation and Conclusions	Various QPs	Various

Section and Title	Qualified Person	Company
26: Recommendations	Various QPs	Various
27: References	Glen Kuntz, P.Geo	Nordmin
28: Glossary	Glen Kuntz, P.Geo	Nordmin

2.3 QP Site Visit

Glen Kuntz, P.Geo., QP, completed a site visit to the Project between April 24 and April 25, 2021. Greg Barnes – Executive VP accompanied Mr. Kuntz and Project geologists Bjorn Jeune and Amber Markan, who collectively have been involved with the Project for multiple years. QP's, Fernando Rodrigues, Daniel Yang, and Klohn Crippen Berger representatives have been to the Project site on multiple occasions in 2019 and previous years in preparation of historical technical reports.

2.4 Effective Dates

The effective date of the Mineral Resource Estimate is November 1, 2021. The effective date of the Technical Report is December 1, 2021.

2.5 Information Sources and References

This Technical Report has been prepared by independent consultants who are QP's under NI 43-101 and prepared in accordance with NI 43-101, Form 43-101F1, and Companion Policy 43-101CP. Subject to the conditions and limitations set forth herein, the independent consultants believe that the qualifications, assumptions, and information used by them is reliable, and efforts have been made to confirm this to the extent practicable. However, none of the consultants involved in this study can guarantee the accuracy of all information in this Technical Report.

This Technical Report is based, in part, on internal company technical reports and maps, published government reports, company letters and memoranda, and public information as listed in Section 27. In addition, several sections from reports authored by other consultants have been directly quoted or summarized in this Technical Report and indicated where appropriate.

Any statements and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of this Technical Report.

The authors of this Technical Report have taken all steps in their professional judgment to verify and confirm the accuracy of the information contained in this report, and other than with respect to these matters set forth in Section 3 hereof, do not disclaim any responsibility for this Technical Report.

2.6 Previous Reporting

The following historical information is relevant to provide context but is not current and should not be relied upon. The QPs responsible for preparing this Technical Report have not done sufficient work to classify the historical estimates as current Mineral Resources or Mineral Reserves, and the Company is not treating any historical estimates as Mineral Resource Estimates.

The Mineral Resource Estimate (effective date of November 1, 2021) discussed herein (Section 14.11) supersedes historical and past Mineral Resource Estimates presented in this section.

- P&E Mining Consultants Inc. (January 6, 2009). Technical Report, Resource Estimate on the Toroparu Gold-Copper Deposit, Upper Puruni River Area, Guyana; NI 43-101 Technical Report No 153, Effective date of October 26, 2008.

- P&E Mining Consultants Inc. (July 26, 2010). Technical Report, Updated Resource Estimate on the Toroparu Gold-Copper Deposit, Upper Puruni River Area, Guyana; NI 43-101 Technical Report No 186, Effective date of May 12, 2010.
- P&E Mining Consultants Inc. (October 13, 2010). Technical Report, Updated Resource Estimate on the Toroparu Gold-Copper Deposit, Upper Puruni River Area, Guyana; NI 43-101 Technical Report No 193, Effective date of September 12, 2010.
- P&E Mining Consultants Inc. (May 5, 2011). Technical Report, Updated Resource Estimate, and Preliminary Economic Assessment of the Toroparu Deposit, Upper Puruni Property, Upper Puruni River Area, Guyana; NI 43-101 Technical Report No. 208, Effective April 30, 2011.
- P&E Mining Consultants, (March 12, 2012). Technical Report, Updated Resource Estimate, and Preliminary Economic Assessment of the Toroparu Deposit, Upper Puruni Property, Upper Puruni River Area, Guyana; NI 43-101 Technical Report No 234, Effective Date of January 30, 2012.
- SRK Consulting (U.S.) Inc. (May 24, 2013). NI 43-101 Technical Report, Prefeasibility Study, Toroparu Gold Project, Upper Puruni River Area, Guyana, No. 349800.020, Effective Date of May 8, 2013, containing the March 31, 2013, Mineral Resource statement.
- SRK Consulting (U.S.) Inc. (June 11, 2019). NI 43-101 Technical Report, Preliminary Economic Assessment Report, Toroparu Gold Project, Upper Puruni River Area, Guyana, No. 349800.100, Effective Date of June 11, 2019, including the September 20, 2018, Mineral Resource statement.

2.6.1 Previous Mineral Reserve Estimates

- SRK Consulting (U.S.) Inc. (2013). NI 43-101 Technical Report, Prefeasibility Study, Toroparu Gold Project, Upper Puruni River Area, Guyana.
- SRK Consulting (U.S.) Inc. (June 11, 2019). NI 43-101 Technical Report, Preliminary Economic Assessment Report, Toroparu Gold Project, Upper Puruni River Area, Guyana, No. 349800.100, Effective Date of June 11, 2019, including the September 20, 2018 Mineral Resource Statement disclosed on September 26 2018 in news release titled "Sandpring Resources Announces Increased Mineral Resources".

2.7 Acknowledgements

Nordmin would like to thank and acknowledge the following people who have contributed to the preparation of this report and the underlying studies under the supervision of the QPs:

Nordmin Personnel

Christian Ballard, P.Geo, Senior Geologist, John McKenzie, Senior Geologist, Annika Van Kessel, GIT, and Sirena Jacobsen, Geological Technician.

GCM Mining Employees and Consultants

Alessandro Cecchi, Exploration VP, Stuart Smith, Senior Metallurgical Engineer and Seit Meka, Senior Process Engineer, Metifex, P. Greg Barnes & Richard Munson, Consultants, & Pascal Van Osta, Geologic Consultant.

2.8 Units of Measure

The following measurement units, formats, and systems are used throughout this Technical Report unless otherwise noted.

- Measurement Units: all references to measurement units use the International System of Units or metric for measurement. The primary linear distance unit, unless otherwise noted, are metres (m).
- General Orientation: unless otherwise stated, all references to orientation and coordinates in this Technical Report are presented as UTM in metres.
- Currencies outlined in the Technical Report are stated in United States dollars (US$) unless otherwise noted.

The symbols and abbreviations used in this Technical Report are outlined in Section 28.4.

3 RELIANCE ON OTHER EXPERTS

Nordmin and the Consultants have assumed and relied on the fact that all the information and existing technical documents listed in the References, Section 27 of this Technical Report, are accurate and complete in all material aspects. While all the available information presented has been carefully reviewed, we cannot guarantee its accuracy and completeness. We reserve the right, but will not be obligated, to revise the Technical Report and conclusions if additional information becomes known after the date of this Technical Report.

3.1 Mineral Tenure, Surface Rights, Property Agreements, and Royalties

Nordmin reviewed copies of the tenure documents, operating licences, permits, and work contracts; independent verification of land title and tenure reported in Section 4, was not performed. Accordingly, Nordmin did not independently verify the legality of any underlying agreement(s) that may exist concerning the licenses or other agreement(s) between third parties but has instead relied on the Company to have conducted the proper legal due diligence.

3.2 Environmental, Permitting, and Liability Issues

The QP has fully relied upon the Company concerning the Project environmental, socio-economic, and permitting matters relevant to the Technical Report.

4 PROPERTY DESCRIPTION AND LOCATION

The Project is located within the Company's 100% controlled Upper Puruni Concession contains 53,283-hectare(s) of mineral leases located in the Cuyuni–Mazaruni Region (Region 7) of Western Guyana, South America (referred to as "the Property"). The Project is comprised of two deposits: the Toroparu Deposit (formerly known as Toroparu or Toroparu Main and Toroparu SE), and the Sona Hill Deposit (formerly known as Sona Hill, Sona Hill prospect, and Sona Hill Gold Deposit). The Toroparu Deposit is located near the main camp. The Sona Hill Deposit is located approximately 5 km to the southeast of the Toroparu Deposit.

4.1 Property Location

The Property is located within the Mazaruni Mining District, one of six mining districts in Guyana. This mining district is located within Region 7 of Guyana, the Cuyuni–Mazaruni Region, one of ten administrative regions within the Country of Guyana. The Property location is shown in Figure 4-1.

The Property is held and operated through ETK, the Company's wholly owned subsidiary.

ETK holds the mineral properties in the Upper Puruni Area. They are comprised of seven Small Scale claims, 65 PPMS, 25 MPs and two contiguous PL that collectively cover an area of 105,802 acres or 42,816.55 ha (Appendix B). The Company will consider further land adjustments as additional exploration work is completed.

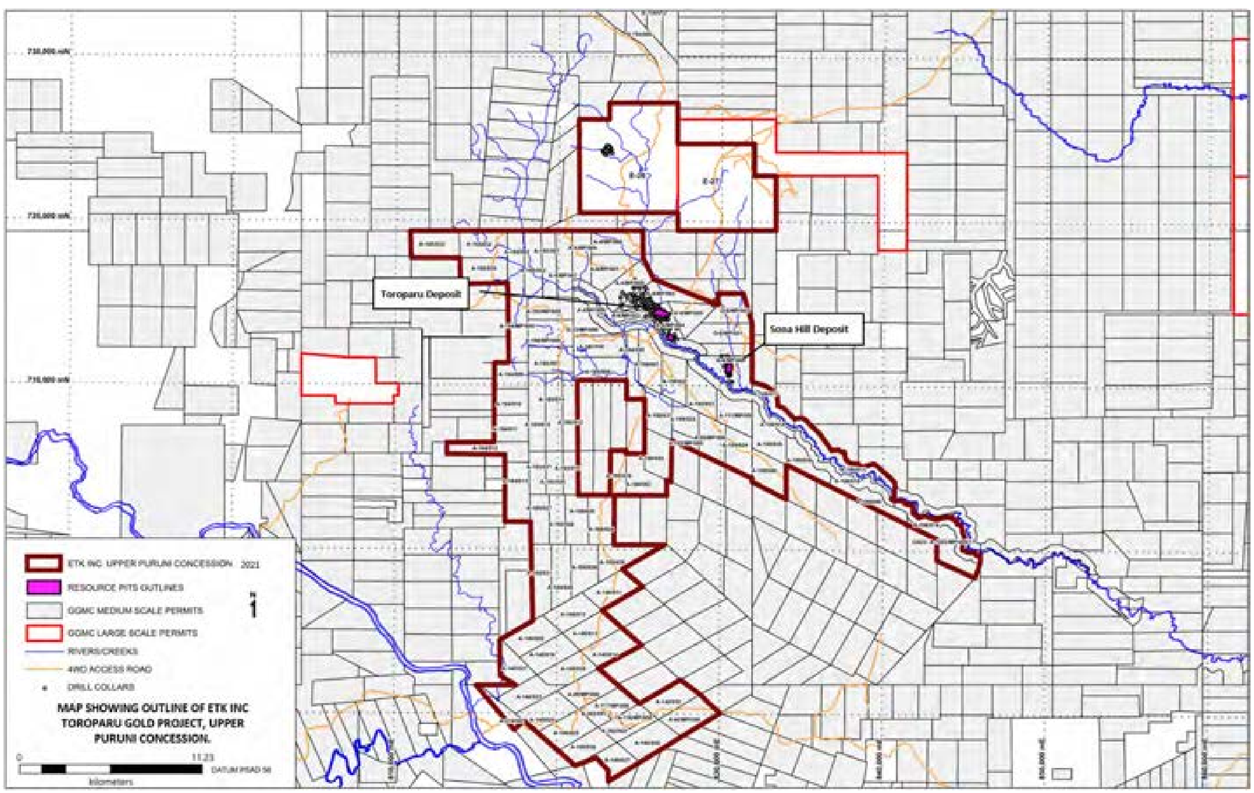


Figure 4-1: Project location map

4.2 Underlying Agreements

The Toroparu Deposit, Main and SE Areas, are located on property subject to the Alphonso Joint Venture. The Sona Hill Deposit is located on property that is subject to the Godette Joint Venture.

4.2.1 The Alphonso Joint Venture

The Alphonso Joint Venture was originally entered into in 1999 and was amended and restated in its entirety in 2008. In March 2020, the Company exercised its purchase option and acquired 100% of the property subject to the Alphonso Joint Venture Agreement. Mr. Alphonso has retained the right to conduct alluvial mining activities on all lands subject to the original Alphonso Joint Venture.

The Alphonso Joint Venture provided that ETK would commence commercial production, defined as the production of 50,000 ounces of gold per year, beginning on January 1, 2013, or, in lieu thereof, pay Mr. Alphonso an annual sum of the Guyana dollar equivalent of US$250,000 until commercial production has commenced. The Company has made all annual payments from January 2013 through December 31, 2019.

The Alphonso Joint Venture further provided that in the event ETK has not achieved commercial production by January 1, 2020, Mr. Alphonso may declare a default under the terms of the agreement. As part of the agreement, the Company had the right to purchase Mr. Alphonso’s entire interest in the Upper Puruni Agreement for US$20 million.

In March 2020, the Company completed the final purchase for Mr. Alphonso’s entire interest and his underlying rights (other than the right to continue alluvial mining) and paid US$20 million ($27,636,600); and paid the balance of US$238,095 ($337,783) in December 2020 as a result of foreign exchange settlement of the payment to Mr. Alphonso.

In December 2019, the Company completed a debenture financing for US$20,000,000, where the proceeds were used in March 2020 to complete the final purchase of the Project.

In July 2020, the Company settled the full principal amount of the Debenture and related accrued interest.

4.2.2 Godette Joint Venture Agreement

Through its wholly owned subsidiary ETK, the Company has rights to three MPs pursuant to the Godette Joint Venture Agreement (the “Godette Agreement”). ETK has sole operatorship and sole decision-making discretion in all matters pertaining to gold exploration on the lands subject to the Godette Agreement. ETK also has the sole and exclusive right to sell all gold, other precious metals, or gemstones it may recover from the properties.

The Sona Hill Deposit is located on the Godette property. ETK purchased 100% of the Godettes’ interest in the Godette Agreement for the sum of US$300,000.

The Godette Heirs remain the registered owner of the Godette MP. However, under the Godette Agreement, the Godette Heirs have irrevocably contributed and committed all their right, title, and interest in the Godette MP for the benefit of ETK and the ETK-Godette Venture and have granted ETK the exclusive right to conduct the ETK-Godette Operations, subject to the rights described above. Further, the Godette Heirs have agreed that during the term of the Godette Agreement, the Godette Heirs will not deal or attempt to deal with any right, title, or interest in the Godette MP or in their interest in the Godette Agreement in any way that would or might affect the right of ETK to conduct the ETK-Godette Operations on the lands subject to the Godette MP.

ETK holds an irrevocable power of attorney from the Godette Heirs, providing ETK, with among other powers, the right to take any action that the Government of Guyana may require to issue a Large Scale Mining Licence covering the three Godette MP’s.

4.2.3 B.M. Mining Agreement

In September 2015, ETK entered into an agreement (the "B.M. Mining Agreement") to acquire the right to explore 25,605 acres of property in the Otomung River area located immediately adjacent to the northwestern boundary ETK's current property block in the Project area.

Under the terms of the final B.M. Mining Agreement, ETK has the option to extend the B.M. Mining Agreement annually by making certain payments to B.M. Mining as described in the B.M. Mining Agreement.

Based on a review of the results of the exploration work done by the Company on the Otomung Block, the Company, during the year ended December 31, 2020, elected not to continue with the B.M. Mining Agreement.

4.2.4 Wheaton Agreement

In November 2013, the Company entered into a precious metals purchase agreement (the "Wheaton PMPA") with Silver Wheaton (Caymans) Ltd., who subsequently changed its name to Wheaton Precious Metals (Caymans) Ltd. (Wheaton). Under the Wheaton PMPA, Wheaton can elect to pay the Company incremental up-front cash payments totalling US$148.5 million for 10% of the payable gold production from the Project.

In addition, Wheaton can elect to make continuing payments to the Company of the lesser of the market price and US$400 per payable ounce of gold delivered to Wheaton over the life of the Project, subject to a 1% annual increase starting after the first three years of production. The Company has received an initial draw down of US$13.5 million of the cash payment, to be used primarily for the advancement of the final feasibility study for the Project.

In April 2015, the Company and Wheaton amended the Wheaton PMPA to include a silver stream under which Wheaton can elect to pay the Company incremental up-front cash payments totalling US$5.0 million for 50% of the payable silver production from the Project. In addition, Wheaton will make ongoing payments to the Company of the lesser of the market price and US$3.90 per payable ounce of silver delivered to Wheaton over the life of the Project, subject to a 1% annual increase starting on the fourth anniversary of production. The Company received an initial draw down of US$2.0 million of the cash payment in four equal instalments over the course of 2015, with the remaining US$3.0 million payable in instalments during construction of the Project.

Under the terms of the Wheaton PMPA, as amended, the Company is required to complete a final feasibility study for its Project before December 31, 2022, upon receipt of which Wheaton can elect to proceed and make the remaining payments totaling US$138.0 million to finance the construction of the Project, or Wheaton can elect to terminate the PMPA.

There are no assurances that Wheaton will elect to fund construction of the Project or that the Company will be successful in securing alternative financing, if available, on terms acceptable to the Company.

4.3 Permits and Authorization

ETK has all the necessary permits and permissions currently required to conduct its exploration work and medium-scale mining and gravity recovery of gold and other minerals on the Project. In addition, the project has its Environmental Authorization, Mineral Agreement, and Fiscal Stability Agreement in place.

4.4 Environmental Considerations

The Project is not the subject of any known environmental liabilities.

4.5 Mineral Properties

Mineral properties in Guyana allow for four scales of operation. These include Small Scale claim licenses of 460 m x 245 m or a river claim consisting of one mile of a navigable river. PPMS and MP cover between 150 acres to 1,200 acres each and are restricted to ownership by Guyanese. However, foreigners may enter into joint venture arrangements whereby the two parties jointly develop the property. PL covering between 500 acres and 12,800 acres are granted to local or foreign companies. Large areas for geological surveys are granted as Permission for Geological and Geophysical Surveys with the objective of applying for PL over the favourable ground.

4.6 Rentals and Royalties

The Company executed a Mineral Agreement with the Government of Guyana (the “Mineral Agreement”) that stipulates a royalty of 8% on gold (1.5% on copper) produced from its mineral properties payable in cash or in-kind to the Government of Guyana.

Mineral properties are also subject to annual rentals. The rental rates for each of the MPs are US$1.00 per acre per annum. Rental rates for each of the PPMS are US$0.25 per acre for the first year, with an increment of US$0.10 per acre for every additional year. Rental rates for PLs are US$0.50 per acre for the first year, US$0.60 per acre for the second year, and US$1.00 per acre for the third year with an increase of $0.50 per acre for the fourth and fifth years.

An application fee of US$100 and a work performance bond equal to 10% of the approved budget are also required. Rentals on the claims/permits controlled by ETK are payable annually by the expiry date of each claim/permit.

Additionally, royalty payments of $20,000,000 payable to the former surface owner at a rate of $2 million per year for 10 years following payback of all initial capital expenditures to ETK Mining are included in the PEA.

The $20 million payments are estimated at a gold price of $1,500 per ounce under an agreement “Surface Owner Royalty Agreement”) executed between ETK Mining and the Alphonso & Sons on November 9, 2011.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Topography, Elevation, and Vegetation

The topography is flat to gently undulating to hilly. Elevations range in the Project area from approximately 94 m to 105 m above sea level (masl) in elevation for the Toroparu Deposit, Main, and Southeast (SE) Areas in relatively flat ground, to 60 masl to 120 masl for the Sona Hill Deposit. The Property is interspersed with steep hills of meta-basic rock up to 200 masl, whereas the metasediments represent flatter topographies. The Project is located in tropical rain forest vegetation, with clearing for the camp, airstrip, and access roads.

The Property is in the deep jungle and is typical of tropical areas with gentle terrain (poorly drained) and high precipitation conditions. The existing saprolite open pit excavation area is at about 93 m in elevation. The current main exploration camp is located at 100 m in elevation.

5.2 Accessibility

The Project has been accessed overland from Georgetown and from Tidewater thru Itaballi on the west bank of the Mazaruni River since 2003. Access to the Upper Puruni Property and the Project by road from Georgetown includes 128 km via paved highway south to Bartica, a ferry crossing of the Essequibo River at Bartica to Itaballi, then 200 km west on a public gravel road to the south gate at Toroparu Junction, and 25 km north to the Project site. Overland travel time is approximately 12 to 16 hours in the dry season (Figure 5-1).

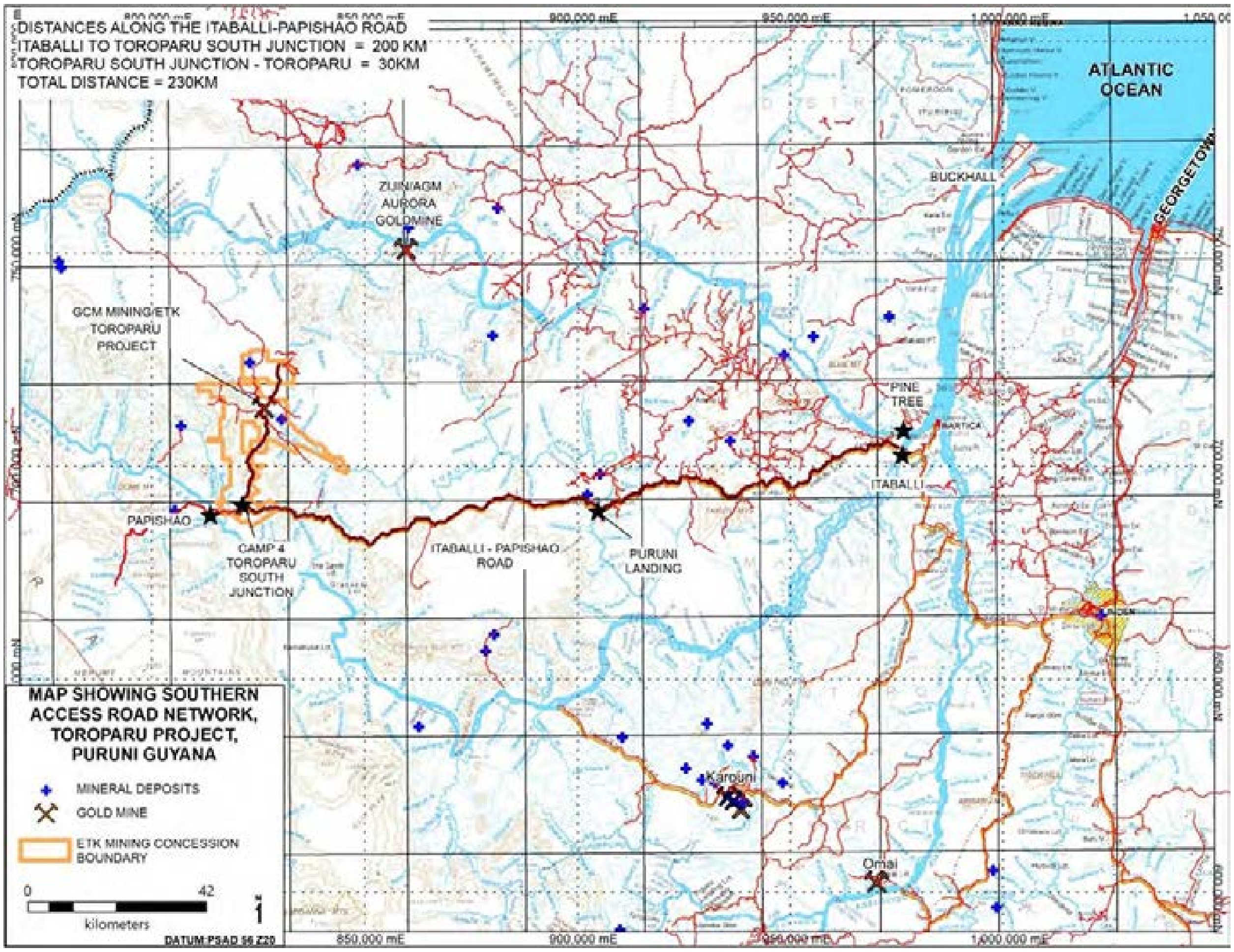
Heavy equipment and cargo are transportable by small, ocean-going vessels and barges on the Essequibo River to Itaballi. There it is loaded on to trucks for the 230 km overland journey to Toroparu crossing the Puruni River at the town of Puruni Landing approximately 60 km from Itaballi on a GCM Mining-ETK operated 40 tonne ferry barge.

This access is being upgraded to serve as the primary access to the Project for construction and mining operations.

Figure 5-1 shows the regional road systems and the Itaballi to the Camp 4-Toroparu South Junction to Toroparu Access Road.

New port facilities at Pine Tree Landing will located on the right bank of the Cuyuni River, which has historically been used by logging companies as a shipping point for lumber products and supplies, will serve as the port connecting Toroparu to tidewater.

The port facility will accommodate ocean-going barges which will transfer cargo between Georgetown and Pine Tree Landing via the Essequibo, Mazaruni and Cuyuni Rivers. Containerized cargo is anticipated to include both imported supplies and copper concentrate for export. Cargo at the port will be handled primarily with 40-ton forklifts and reach stackers. Heavy or oversized cargo will be handled by a mobile harbor crane or as roll-off cargo.



Source: ETK, 2021

Figure 5-1: Project access

The Itaballi-Papsihao Road has been used to access the Project since 2003. The Upper Puruni Concession entry gate is located at Toroparu South Junction, 25 km due south of the Project on a private access road. Access to the Upper Puruni Property and the Project by road from Georgetown includes 128 km via paved highway south to Bartica, a ferry crossing of the Essequibo River at Bartica to Itaballi, then 200 km west on a public gravel road to the south gate at Toroparu Junction, and 25 km north to the Project. Overland travel time is approximately 12 hours to 16 hours in the dry season from August to May. Heavy equipment and cargo are transportable by small, ocean-going vessels, and barges on the Essequibo River to Itaballi. The heavy equipment and cargo are loaded onto trucks for the 230 km overland journey to Toroparu crossing the Puruni River at the town of Puruni Landing approximately 60 km from Itaballi on a GCM Mining-ETK operated 40 tonne ferry barge.

5.3 Climate

Climate data consists of temperature, precipitation, and humidity data available from the meteorological station at the Project which was established during the baseline environmental studies in 2007, and subsequent data compiled by KCB (with record periods of up to five years), supplemented with historical records from two regional stations managed by the Guyanese Hydrometeorological Agency (Mazaruni [200 km from the Project] and Enachu [75 km from the Project]) stations.

The average temperature on site is estimated to be 28ºC, with minimum and maximum recorded temperatures of 18ºC and 38ºC, respectively, as estimated from the Toroparu Station. Humidity is relatively high with values ranging between 64% and 100% and an average value of 82%.

Monthly average rainfall values have been estimated from the data obtained from the Toroparu and Enachu stations. The precipitation records gave an average annual precipitation of 2,625 mm. A wet season occurs in December to February and a second wet period in May to July of each year. Although mining operations can be carried out on a year-round basis, the dry season from July to November is the most advantageous time to carry out exploratory surveys such as geochemical sampling, drilling, and geophysical surveys.

5.4 Sufficiency of Surface Rights

The Company has sufficient surface rights through holding the mineral properties as further expanded by rights granted to acquire additional property under the terms of the Mineral Agreement.

5.5 Infrastructure

The town of Bartica, with a population of approximately 11,000, is approximately 230 km from the site. Georgetown is 385 km away by road. There are some small temporary mining camps along the access road to the Project site.

The on site infrastructure includes the airstrip, on site service roads, river crossings, camp water supply and treatment, generator-set power supply for the camp, and the main camp facilities. Site access roads, which interconnect the various site service areas are segregated to the maximum extent possible from the mine haul roads.

Equipment and supplies for construction and mining will come from Georgetown.

5.5.1 Power

There is no nearby electricity grid. Therefore, permanent power will not be available at the Project before the completion of construction. Construction will rely wholly on power from temporary thermal power generators. Permanent power will be generated on site by a hybrid thermal-solar PV power generation station built, owned, and operated (BOO) by a third party independent power producer.

5.5.2 Water

Water is available throughout the year from the Wynamu River, its creeks, and rainfall run-off.

5.5.3 Mining Personnel

Labourers with various experience in heavy equipment operation are available in Georgetown, Linden, and from other population centres in Guyana.

5.5.4 Tailings Storage Areas

KCB investigated several potential TSF sites in 2012. Five sites were evaluated, compared, and ranked. The preferred site is the currently defined TSF location lying approximately 9.5 km to the northeast of the Project site. The site was the subject of a feasibility level study completed in 2016. The study defines three modules with a total nominal capacity of 156 million tons, of which the mine plan defined in this PEA will deposit 107 Million tons over a 24 year mine life.

5.5.5 Waste Disposal Areas

The PEA design includes two waste rock dumps located in proximity to the Toroparu and SE open pits. The rock dumps will be underlain by waste saprolite on top of which waste fresh rock will be dumped in 7 m lifts.

The East Dump will be located directly east of the final Toroparu Pit and north of the SE Pit, and the North Dump directly north of the NW Toroparu Pit. The Sona Hill Waste Dump is located west of the Sona Hill Pit. Depending on future mine scheduling, some waste material may be placed in to be mined open pits after completion of mining (such as the SE Deposit proposed pit).

5.5.6 Processing Plant/Ore Storage/Mine Facilities Pad

The process mill, power generation, mine maintenance and administration facilities are located approximately 1 km northeast of the Toroparu Mine Area at an elevation approximately 20 m above the Wynamu River that flows between the mining and milling operations.

5.5.7 Permanent Camp

A permanent camp will be constructed approximately 2 km east of the processing plant/administration complex behind a hill and upwind of the mill processing and administration facilities. The camp includes kitchen, dining, recreational, and medical and camp administration facilities accessible by vehicle and bus from the main facilities area.

6 HISTORY

6.1 Prior Ownership, Exploration, and Development Work

The Company (at the time known as Sandspring Resources Ltd. [Sandspring]) was incorporated pursuant to the provisions of the Business Corporations Act (Alberta) on September 20, 2006 and continued into the province of British Columbia in November 2019. On November 24, 2009, Sandspring announced the completion of the acquisition of 100% of the issued and outstanding shares of GoldHeart Investment Holdings Ltd. (GoldHeart). GoldHeart, through its wholly owned subsidiary ETK Inc. (ETK), holds mineral and prospecting rights to the Toroparu Gold project and adjacent properties, collectively known as the Upper Puruni Concession. On November 29, 2019, Sandspring changed its name to Gold X Mining Corp (Gold X). Gold X was acquired by Gran Colombia Mining Corp. (now GCM Mining Corp.) in June of 2021.

Table 6-1: Previous Project Exploration and Mining, 1880 to 2009

Activity
Late 1880's to early 1900's
Historical exploitation of alluvial gold and diamonds in the Toroparu area dates back to about 1887. Conolly (1926) described alluvial diamond operations up to about 1914, to the northwest of the Toroparu area. Grantham (1934) described gold and diamond workings in the Majuba Hill and Wynamu areas. The Wynamu River lies adjacent to the Toroparu saprolite open pit. It is labelled as “Toroparu River” on some older maps.
1950
Pollard and Hamilton created a geological map of the area on which the locations of gold and diamond workings were noted (Pollard, 1950).
1997
Alphonso commenced alluvial mining at Toroparu, mining old tailings and river alluviumby washing material into a pit with high pressure water jets and pumping the slurry up to a sluice box. By 1999, much of the alluvial material was exhausted, and work proceeded deeper into the underlying saprolite. Thus, the surficial alluvial area was gradually developed into a pit (the Toroparu saprolite open pit). The Alphonso operations continued until 2001.
1999
ETK began auger drill sampling to the east and west of the pit area and evaluated the possibility of re-working the tailings. Reports by Hopkinson (1999), Uzunlar (2000) and Shaffer (2000, 2001, and 2003) summarize the available assay data.
2000
The GGMC carried out regional mapping and geochemical drainage sampling **(Heesterman, Kemp, & Nestor, 2001)** that showed an anomalous gold and copper area in the immediate Toroparu area.
ETK entered into an exploration joint venture with Alphonso and commenced rehabilitation and upgrading of the 240 km access road into the Property to facilitate the transport of mining equipment and supplies to the mine site.
2001

Activity
Alphonso ceased mining operations in the Toroparu saprolite open pit. 15 “land dredges” were employed at the peak of the Alphonso mining activity in the Toroparu saprolite open pit area. It has been estimated that 60,000 ounces of gold may have been produced historically over a 70 – year period from the Toroparu area.
2001 to 2003
ETK carried out further auger drill sampling to the east and west of the Toroparu saprolite open pit area. This work reportedly identified an estimated 1.4 million tons of historic auriferous tailings located southeast of the main pit area.
2003 to 2004
Heesterman carried out drainage geochemical sampling in the PL blocks located north of the Toroparu saprolite open pit and around the pit itself, reporting that gold mineralization could extend at least 6 km to the northwest and 1 km to the southeast of the Toroparu saprolite open pit.
2004
ETK commissioned a gravity circuit to test-mine the gold-bearing tailings and saprolite and also conducted exploration for additional gold sources defined in the GGMC regional geochemical and prospecting survey of the Upper Puruni Area.
From December 2004 to April 2007 ETK conducted intermittent, seasonal test-mining from saprolite, in the Toroparu saprolite open pit using a combination of hydraulic sluicing and a gravity circuit with screens, ball mill, Falcon centrifugal concentrators, and shaker tables.
2005
In 2005 and 2006, two phases of trench-channel sampling were completed by Meixner and Wesa to investigate the gold mineralization in the saprolites of the pit area and to determine the suitability for conducting further exploration. A zone of gold mineralization, over an area of about 180 m x 100 m, was identified in the saprolitic rock of the pit area with average grades in the general range of 0.5 g/t Au to 1.5 g/t Au. This zone was open in all directions.
2006 & 2007
TerraQuest conducted a 5 km x 4.5 km high resolution Tri-sensor Magnetic and Radiometric Airborne Survey around the Toroparu saprolite open pit area in October 2006. The pit area was found to lie within a magnetically low area just to the south of a large magnetic high area of unknown provenance. The survey outlined a number of magnetic and radiometric anomalies in the areas adjacent to the Toroparu saprolite open pit.
ETK initiated the Phase I drill program in December 2006 as recommended by Meixner and Wesa (Meixner and Wesa, 2006). Phase I included the drilling of 13 NQ cored drill holes (3,480 m) under and around the Toroparu saprolite open pit by March 2007. Phase II drilling of an additional 10 NQ cored drill holes (3,748 m) was completed in August 2007. Phase I and II drilling defined a mineralized block of 600 m x 300 m x 300 m around the Toroparu saprolite open pit (Meixner 2008).

Activity
2008
The ETK Phase III drill program consisting of 6 NQ cored drill holes (2,590 m) was undertaken from April to May 2008. A total of 30 drill holes (TPD001-030) comprising 10,218 m defined a zone of mineralization of 650 m x 350 m x 425 m. That was open in all directions. Twenty seven holes totalling 9,492 m formed the basis for the initial Mineral Resource Estimate published in P&E Mining Consultants Inc's Technical Report No 153, with an effective date of October 26, 2008.
ETK carried out additional auger drill sampling to the northwest of the pit area over a 2 x 3 km area, using a mechanized auger. Nine north easterly lines of auger samples, spaced 500 m apart, were sampled to 5 m depths at approximately 50 m sample intervals. This survey tested the saprolitic rocks beneath the alluvial cover for gold and copper in an area of historic gold workings that lies to the northwest of the Toroparu saprolite open pit.
2009
Approximately 2,500 saprolite samples were collected using hand and power augers during 2009 to depths from 1 m to 15 m. The soil grids were oriented perpendicular to regional structures, extending approximately 4.5 km to the west-northwest (WNW) from the Toroparu resource area. Assay results show several areas of gold enrichment along trend with the highest assay value equal to 9.94 g/t Au.
Forty one trenches totalling 6,000 m spaced at regular intervals and oriented perpendicular to the regional structural trend where possible were completed from August to October along a 5 km strike length to the northwest of the Toroparu saprolite open pit.
Throughout 2009 an initial metallurgical scoping test program was conducted on saprolite and hardrock samples, collected from drill core in the Toroparu Deposit. The goal of this program was to scope the amenability of the saprolite and bedrock for typical gold extraction and copper recovery methods.
Sandspring acquired 100% of GoldHeart, who owned 100% of ETK's outstanding stock, in a transaction that closed on November 24, 2009.

Table 6-2: Project Exploration, 2010 to 2019

Activity
2010
During the first phase of the 2010 drilling program, the Company carried out resource definition and infill drilling of the Main Zone and step out drilling east-southeast and west-northwest of the Toroparu saprolite open pit to upgrade the resource categories of the mineralization encountered in previous drill programs. The second phase, 27 diamond drill holes, totalling 15,844 m, was focused on the west part of the Main Zone. Most of the holes were drilled along trend to the northwest and up to approximately 1,400 m from the western outline of the previous contour of the Indicated resources. In the last quarter five geotechnical holes, totalling 2,303 m, were drilled for pit slope geotechnical design purposes.
Gradient Array Induced Polarization (IP) and Magnetometer surveys were performed over the Toroparu Deposit area and recon grids over Ameeba, Manx, and Timmermans prospects. A total of 85 line-km was completed.

Activity
CM Power undertook an evaluation of the hydro power capacity of the Kumarau Falls (a former United Nations Development Program sponsored project) on the Kumurung River, located 35 km southwest of the Toroparu site. Hydroelectrical Project.
Two Mineral Resource updates were completed: • P&E Mining Consultants Inc. (July 26, 2010). Technical Report, Updated Resource Estimate on the Toroparu Gold-Copper Deposit, Upper Puruni River Area, Guyana; NI 43-101 Technical Report No 186, Effective date of May 12, 2010. • P&E Mining Consultants Inc. (October 13, 2010). Technical Report, Updated Resource Estimate on the Toroparu Gold-Copper Deposit, Upper Puruni River Area, Guyana; NI 43-101 Technical Report No 193, Effective date of September 12, 2010.
2011
A resource definition core drilling program, for a total of 120 drill holes totalling 42,320 m was completed. The program focused on the eastern main mineralized zone of the Toroparu Deposit to increase the overall resources and the average grades of gold and copper and the conversion of resources from the Inferred to Measured and/or Indicated categories. At the end of 2011, and over six years (December 2006 to December 2011), a total of 111,668 m of resource definition drilling was realized in 225 core holes.
A metallurgy gold deportment study was carried out by SGS on a 400 kg composite sample of the Toroparu mineralization and collected in 23 different core holes. Results were received in May 2011. The objective of this investigation was to determine the occurrence of gold, including microscopic and sub microscopic gold in the sample, and to identify, and evaluate any mineralogical factors that might affect potential gold recoveries.
A step out exploration core drilling program totalling 78 core holes for 24,834 m was drilled in adjacent zones northwest and southeast of the Main Zone area to explore for significant and economic extensions of the resource or nearby satellite deposits.
Small recon core drilling campaigns were carried out over areas with promising surface exploration results; including the Ameeba and Manx areas, located respectively at several kilometres northwest and northeast of the Toroparu Deposit. A total of 28 holes for 8,405 m were completed.
A regional saprolite geochemistry sampling campaign was started in March 2011. The survey was focused on areas with presumed geological potential for gold. Semi-regional and detailed geochemical sampling was performed on areas where alluvial mining activities showed gold potential. At the end of 2011, a total of 4,390 samples were collected.
Combined Gradient Array IP and Magnetometer surveys were carried out over several golf prospects, including Ameeba, Timmermans, Manx, and NW of the Toroparu Deposit, completing the grids of the 2010 surveys. An additional 17 line-km of survey were completed.
During the second quarter, a LIDAR survey was flown over an area of 250 km² around the Toroparu Deposit. A detailed topographic contour map was produced from this data.
In June, a project was undertaken to improve and rehabilitate the access road to the Toroparu site (a total distance of 240 km). This work was carried into 2012 when a road work contract was signed with the GGMC to finance part of the total road rehabilitation costs. Portions of the work (+/-100 km) was subcontracted to Mekdeci Machinery and Construction (MMC), a local construction company.

Activity
A Mineral Agreement was signed in November between the Government of the Republic of Guyana and the Company. The Mineral Agreement defined all fiscal, property, import-export procedures, taxation provision, and other related conditions for the continued exploration, mine development and mining/processing operations at the Project. Furthermore, the Government of Guyana agreed to grant a Large Scale Mining Licence, which allows the start of commercial production, once economic feasibility is demonstrated.
Drilling through an effective date of December 31, 2010, was incorporated into P&E Mining Consultants Inc.'s NI 43-101 Technical Report titled Updated Resource Estimate and PEA of the Toroparu Deposit, Upper Puruni Property, Upper Puruni River Area, Guyana, effective April 30, 2011.
A PFS Pit Slope Design report was provided in October 2011.
2012
During 2012 a total of 34,055 m, in 142 holes were completed for the Toroparu Deposit area. At the end of 2012, and over six years (December 2006 through December 2012), a total of 145,723 m of resource definition drilling was completed in 367 holes.
Regional and detailed saprolite hand auger sampling and testing concerning regional gold potential and local gold anomalous zones were conducted. A total of 3,480 samples were collected. Throughout 2011 and 2012, the geochemical surveys covered around 450 km² and 7,850 geochemical samples were collected.
Re-analyses of the 2008 airborne geophysical survey data were carried out. The work consisted of a basic structural interpretation of the aeromagnetic and radiometric data, and an attempt to characterize the geophysical signature of the Toroparu Deposit. This study contributed significantly to the development of a regional exploration model.
FMG completed a Preliminary Design Study in March on the access road reconstruction from the Itaballi port facility to the Toroparu site (230 km); including a conceptual roadway reconstruction design plan, cost estimates, and preliminary solicitation of qualified contractors.
In June, the Environmental Permit was signed and granted by the Environmental Protection Agency.
In September, a monitoring program to assess Kumurung river flow characteristics upstream and downstream and rain fall measurements in the Project watershed commenced.
The culmination of the work programs from the inception of the land agreement in 2009 through to October 2011 was the basis of P&E Mining Consultants Inc.'s NI 43-101 Technical Report titled Updated Resource Estimate and PEA of the Toroparu Deposit, Upper Puruni Property, Upper Puruni River Area, Guyana, effective January 30, 2012.
During 2012 a total of 34,055 m, in 142 holes were completed for the Toroparu Deposit area. At the end of 2012 and over six years (December 2006 through December 2012), a total of 145,723 m of resource definition drilling was completed in 367 holes.
2013
SRK presented a Mineral Resource statement and an estimate of Mineral Reserves, as part of a Prefeasibility Study presented in an NI 43-101 Technical Report titled "NI 43-101 Technical Report, Prefeasibility Study, Toroparu Gold Project, Upper Puruni River Area, Guyana," dated May 24, 2013.

Activity
The PFS modelled an open pit mine with a Proven and Probable Mineral Reserve containing 4.1 Moz of gold and 211 Mlbs of copper, which in contained gold terms represented 60% of the 6.9 Moz (in resource – pit shell) Measured and Indicated Mineral Resource Estimate.
2013 – 2018
Following the 2013 Prefeasibility Study, the Company continued to conduct exploration to evaluate other areas on the Property, with basic exploration techniques such as auger sampling and soil sampling to follow up on target areas defined by regional structure or geology, past or present alluvial gold prospecting, and know gold occurrences, and by exploration drilling, including: • Regional geochemical sampling and auger sampling of saprolite at the Otomung concession, and infill saprolite sampling at Sona Hill in advance of drilling. • Three campaigns of exploration core drilling at the Sona (2015, 2016, and 2017), 184 angle core holes for a total of 21,963 m, culminating with a Mineral Resource Statement effective September 20, 2018[1]. • Exploration drilling of 62 shallow core holes for 6,433 metres on the Wynamu exploration target**.**
The Government of Guyana completed its review of the Company's work on the Kurupung River Hydroelectric Project located at Kumurau Falls, approximately 50 km southwest of the Project. on August 6, 2018, following the review, the Company and the Government signed an Amended and Restated Memorandum of Understanding granting the Company exclusive rights for hydroelectric development through December 31, 2021.
2019
June 11, 2019, the Company released the SRK Consulting (U.S.) Inc. NI 43-101 Preliminary Economic Assessment Technical Report of the Toroparu Gold Project, Upper Puruni River Area, Guyana.

6.2 Historical Mineral Resource Estimates

The Mineral Resource Estimate (effective date of September 20, 2018) discussed herein (Section 14.9) supersedes historical and past Mineral Resource Estimates.

The following historical information is relevant to provide context but is not current and should not be relied upon. The QPs responsible for preparing this Technical Report have not done sufficient work to classify the historical estimate as current Mineral Resources or Mineral Reserves, and the Company is not treating any historical estimates as Mineral Resource Estimates.

- P&E Mining Consultants Inc. (January 6, 2009). Technical Report, Resource Estimate on the Toroparu Gold-Copper Deposit, Upper Puruni River Area, Guyana; NI 43-101 Technical Report No 153, Effective date of October 26, 2008.
- P&E Mining Consultants Inc. (July 26, 2010). Technical Report, Updated Resource Estimate on the Toroparu Gold-Copper Deposit, Upper Puruni River Area, Guyana; NI 43-101 Technical Report No 186, Effective date of May 12, 2010.

[1] Announced in the Sandspring news release dated September 26, 2018, titled *Sandspring Resources announces Increased Mineral Resources* and is available on SEDAR.com. This news release was not material therefore a technical report was not prepared.

- P&E Mining Consultants Inc. (October 13, 2010). Technical Report, Updated Resource Estimate on the Toroparu Gold-Copper Deposit, Upper Puruni River Area, Guyana; NI 43-101 Technical Report No 193, Effective date of September 12, 2010.
- P&E Mining Consultants Inc. (May 5, 2011). Technical Report, Updated Resource Estimate, and Preliminary Economic Assessment of the Toroparu Deposit, Upper Puruni Property, Upper Puruni River Area, Guyana; NI 43-101 Technical Report No. 208, Effective April 30, 2011.
- P&E Mining Consultants, (March 12, 2012). Technical Report, Updated Resource Estimate, and Preliminary Economic Assessment of the Toroparu Deposit, Upper Puruni Property, Upper Puruni River Area, Guyana; NI 43-101 Technical Report No 234, Effective Date of January 30, 2012.
- SRK Consulting (U.S.) Inc. (May 24, 2013). NI 43-101 Technical Report, Prefeasibility Study, Toroparu Gold Project, Upper Puruni River Area, Guyana, No. 349800.020, Effective Date of May 8, 2013, containing the March 31, 2013, Mineral Resource statement.
- SRK Consulting (U.S.) Inc. (June 11, 2019). NI 43-101 Technical Report, Preliminary Economic Assessment Report, Toroparu Gold Project, Upper Puruni River Area, Guyana, No. 349800.100, Effective Date of June 11, 2019, including the September 20, 2018, Mineral Resource statement.

6.3 Historical Mineral Reserve Estimate

The following historical information is relevant to provide context but is not current and should not be relied upon. The QPs responsible for preparing this Technical Report have not done sufficient work to classify the historical estimate as current Mineral Resources or Mineral Reserves, and the Company is not treating any historical estimates as Mineral Reserve Estimates.

- SRK Consulting (U.S.) Inc. (May 24, 2013). NI 43-101 Technical Report, Prefeasibility Study, Toroparu Gold Project, Upper Puruni River Area, Guyana, No. 349800.020, Effective Date of May 8, 2013.

The PFS modelled an open pit mine with a Proven and Probable Mineral Reserve containing 4.1 Moz of gold and 211 Mlbs of copper, which in contained gold terms represented 60% of the 6.9 Moz (in resource – pit shell) Measured and Indicated Mineral Resource Estimate.

6.4 Past Production

Historical mineral production from the Project has been limited to shallow alluvial and saprolite mining and placer processing of material for free gold. Nordmin noted small scale sluice and riffle table processing of alluvium, and shallow lag gravels, is ongoing to a minor extent at the time of the site visit in 2021. Records of placer production are lacking.

Previous consultants have estimated that perhaps as much as, but not likely more than 100,000 oz of gold has been produced to date for all the historical placer production in the Project area.

In 2004 ETK commissioned a gravity circuit to test-mine the gold-bearing placer tailings and saprolite and conducted exploration for additional gold sources defined in the GGMC regional geochemical and prospecting survey of the Upper Puruni Area.

From December 2004 to April 2007, ETK conducted intermittent, seasonal test-mining from saprolite, in the saprolite open pit using a combination of hydraulic sluicing and a gravity circuit with screens, ball mill, Falcon centrifugal concentrators, and shaker tables.

During this period, an estimated 5,000 oz of gold were produced by ETK from the saprolite material located in the historical saprolite open pit area and alluvial workings in the surrounding areas.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Guiana Shield, the northern half of the Amazonian Craton, underlies the eastern part of Venezuela, Guyana, Surinam, French Guyana, and parts of northern Brazil (Figure 7-1). It is also among the least documented of Precambrian terranes due to thick weathering profiles, tropical vegetation, and tertiary sands (Voicu, Bardoux, & Stevenson, 2001). This region is bound in the north by the Atlantic Ocean and the south by the Amazon-Solimoes basin. There are two undisputed terranes in the Guiana Shield, the Imataca Complex in northwestern Venezuela and the Trans-Amazonian granitoid-greenstone belts in the easternmost extension in Amapá, Brazil (Figure 7-1). The older of the two terranes include the orogenic Archean and Paleoproterozoic sequences represented by the Imataca Complex basement rocks, which have been whole rock dated using U-Pb to 3400 Ma to 3700 Ma. The Imataca Complex comprises gneisses, migmatites, and volcano-sedimentary rocks at granulite or amphibolite facies. The NE-SW trending Guri Fault juxtaposes gneissic rocks of the Imataca against Paleoproterozoic volcano-plutonic terrances of the Pastora Supergroup and the Supamo Complex in Northern Venezuela (Voicu, Bardoux, & Stevenson, 2001).

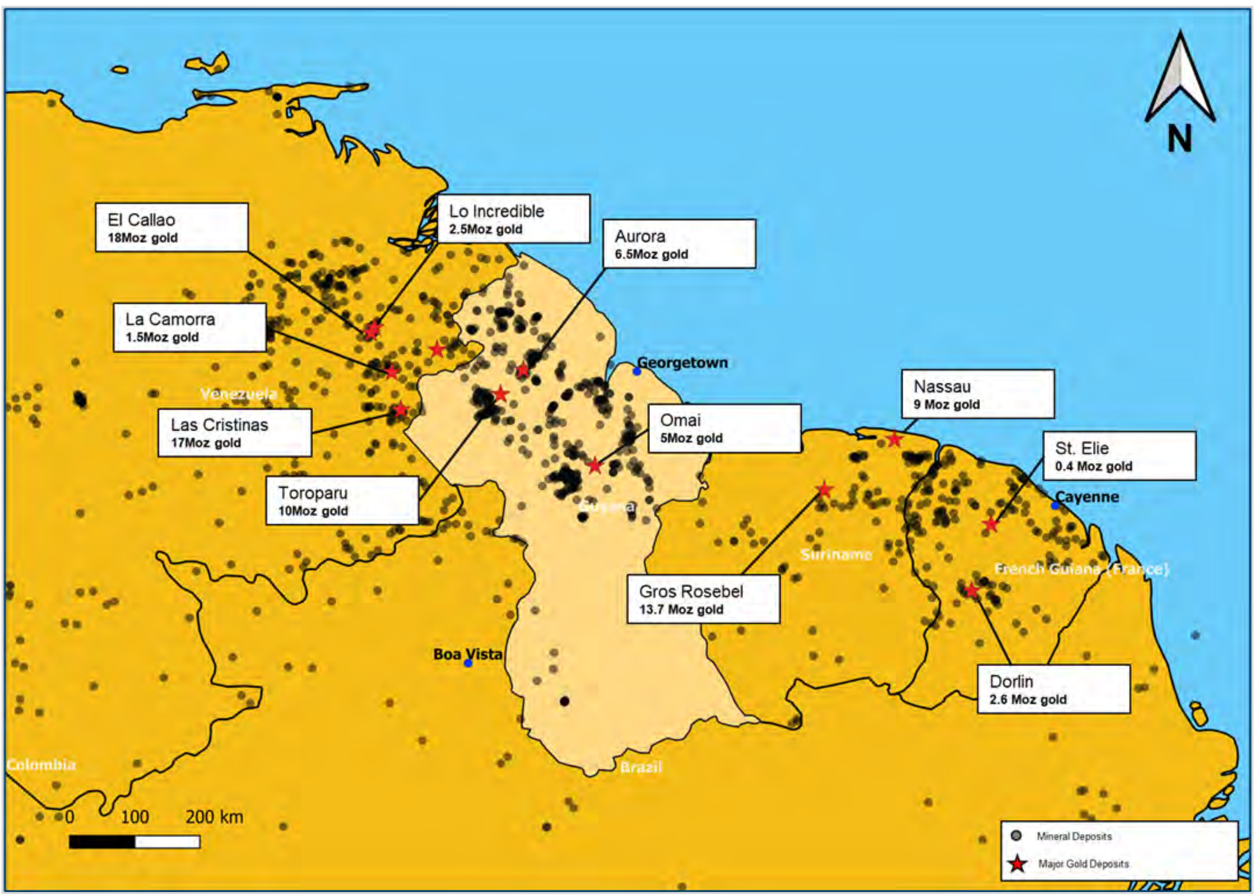


Figure 7-1: Locations of mineral deposits and significant gold deposits within the Guiana Shield in South America

Paleoproterozoic low-grade volcano-sedimentary greenstone sequences give ages of 2250 Ma and 2110 Ma (U-Pb and Sm-Nd whole rock dating, respectively (Voicu, Bardoux, & Stevenson, 2001)). Some lower grade, greenschist facies occur in Suriname and French Guiana.

Voluminous granitoid intrusions throughout the shield were classified as syn- to post-tectonic. The stratigraphy shows a succession of tholeiitic mid-ocean-ridge and back-arc basin basalts, often with pillow selveges; this is followed by more evolved, island-arc type andesites, dacites, rhyolites, and intercalated chemical sediments. This is then followed by sequences of greywackes and shales and finally overlain by epicontinental fluvial deposits. These were formed during the Trans-Amazonian Orogeny, mainly between 2.26-1.98Ga (Gibbs & Barron, 1993) (Avelar, et al., 2003) (Kroonenberg, Mason, Kriegsman, Wong, & De Roever, 2019)). Large diabase dykes, sills and sheets of the Avanavero Suite intrude the Trans-Amazonian terranes (Voicu, Bardoux, & Stevenson, 2001).

In Guyana, the Barama-Mazaruni Supergroup comprises three subparallel northwest-oriented belts (Voicu, Bardoux, & Stevenson, 2001). These belts consist of theoleiitic basalts, gabbros, hornblendites, minor komatiites, calc-alkaline andesitic flows and terrigeneous sedimentary rocks (Gibbs, Montgomery, O'Day, & Erslev, 1986). Greywacke in the northern belt is dated to 2250 ± 106 Ma (U-Pb zircon; (Gibbs & Olszewski Jr., 1982), while a mafic volcanic rock from Omai in the southern belt has been dated with an Sm-Nd isochron age of 2171 ± 140 Ma (Voicu, Bardoux, Harnois, Stevenson, & Crepéau, 1997). The subparallel belts have been interpreted to have formed by successive back-arc closure and extensional oceanic-arc systems caused by migrating spreading ridges. Granitoid intrusions in Guyana occupy large areas within the greenstone sequences and are generally referred to as the "Granitoid Complex" (Voicu, Bardoux, Harnois, Stevenson, & Crepéau, 1997) (Voicu, Bardoux, & Stevenson, 2001).

Large scale ductile shear zones have not been well documented in the Guiana Shield (Voicu, Bardoux, & Stevenson, 2001). In Northcentral Venezuela, the most well documented structure has been the NE-SW tending Guri Fault. A kilometre wide, WNW-ESE striking deformation zone has been identified by geophysical and satellite imagery, this is the Central Guiana Shear Zone (CGSZ) extending westerly toward central Suriname and further west toward northcentral Guyana. The CGSZ is an important area of gold exploration, and several gold discoveries have been found off splays from this main deformation corridor (i.e., Omai, Aurora, Las Cristinas, Lo Incredible) (Figure 7-2) (Voicu, Bardoux, & Stevenson, 2001). Brittle faults are common throughout the Guiana Shield, defining conjugate patterns that are either north-northwest (NNW)-SSE or NNE-SSW.

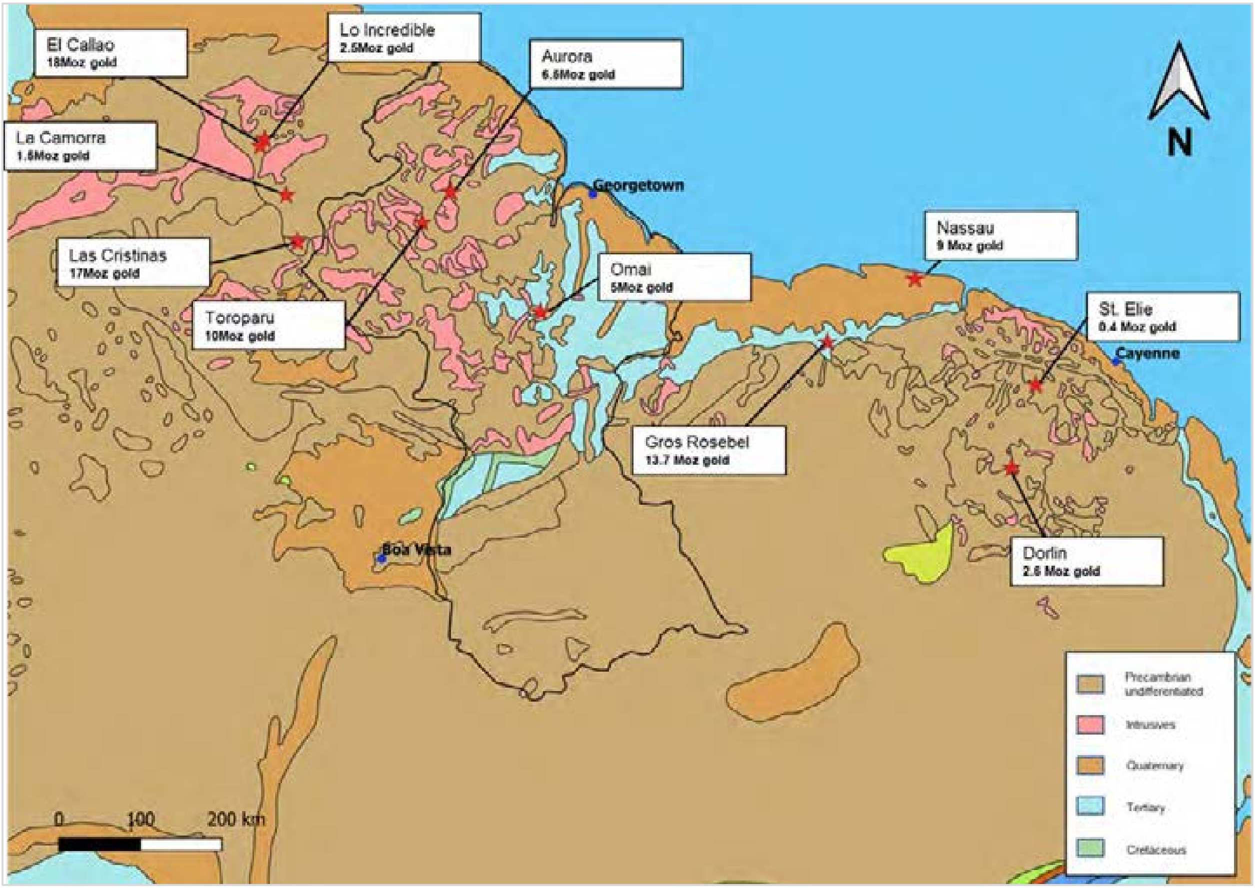


Figure 7-2: Regional geology of the Upper Puruni River Area, Guyana

The principal metallogenic features of gold deposits in the Guiana Shield include being in supracrustal terranes metamorphosed from prehnite-pumpelleyite to upper-greenschist facies. Lithological sequences are sub vertical due to folding/thrusting. Granitoid hosted gold mineralization can be either deformed or undeformed. All deposits post-date peak metamorphism but may be contemporaneous with shearing or syn- to post-deformation. Host lithologies are variable, and proximity to intrusives is common but not essential. Gold deposits are strongly associated with quartz veining and to a lesser extent stockworks, breccias, and lenses. There is a strong correlation between gold deposits and NW-SE striking major shear zones.

The northeastern half of the Upper Puruni Concession, where the Project is located, is underlain by thick volcano-sedimentary sequences consisting of alternating mafic, intermediate, and to a lesser extent, felsic volcanic flows, and pyroclastics, with intercalated sedimentary successions, generally metapelites, and greywackes. These formations form the Puruni VS belt, which extends in a northwesterly direction in between two large plutonic areas, the Aurora batholith located to the northeast of the concession, and the Putareng batholith underlying most of the southwestern part of the Property (Figure 7-3 and Figure 7-4).

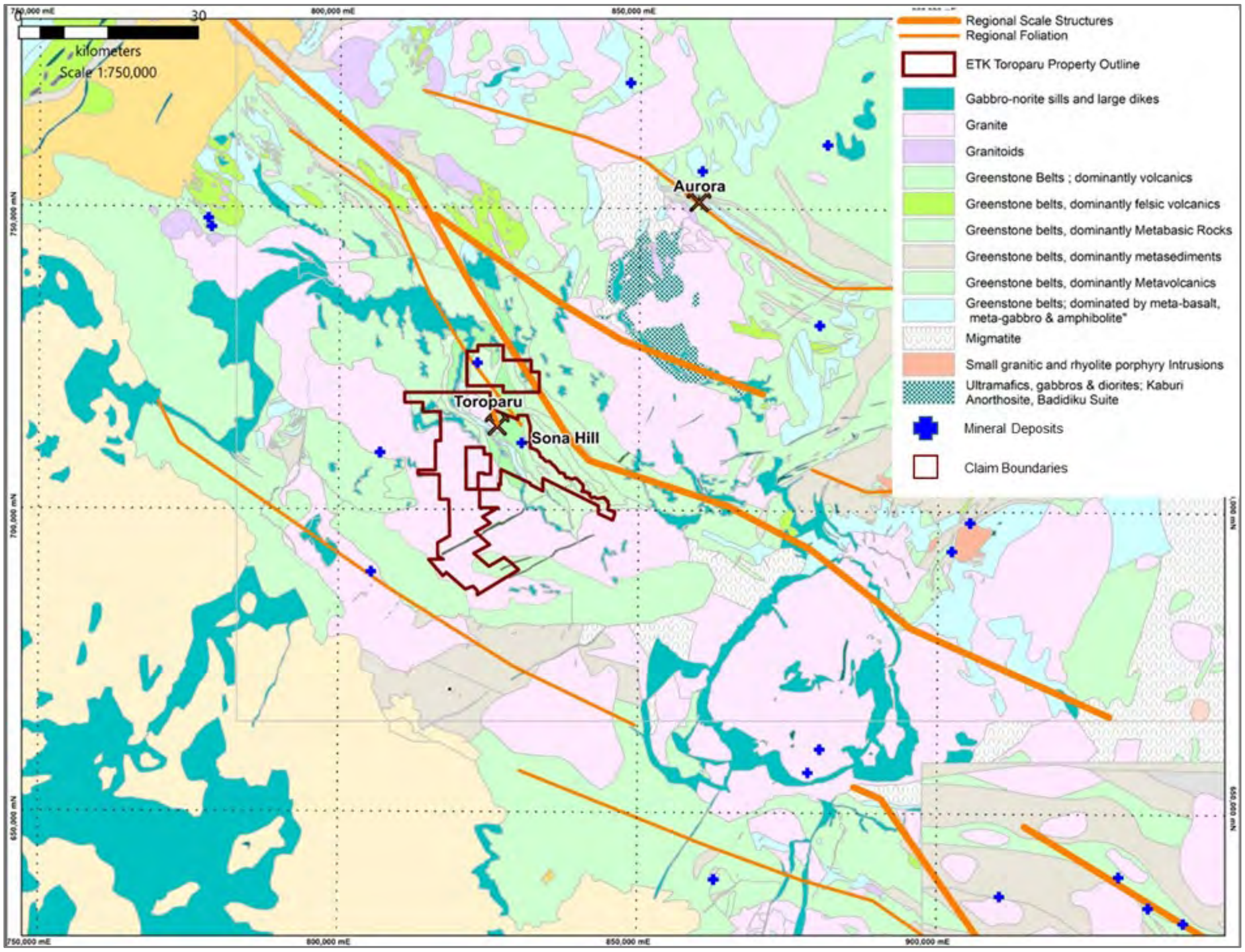


Figure 7-3: Company claim map overlain on regional, local, geology showing major deformation corridors in orange. Blue crosses represent other mineral deposits. A strong association is seen with mineral deposits and major regional scale structures and contacts between VS sequences and granites.

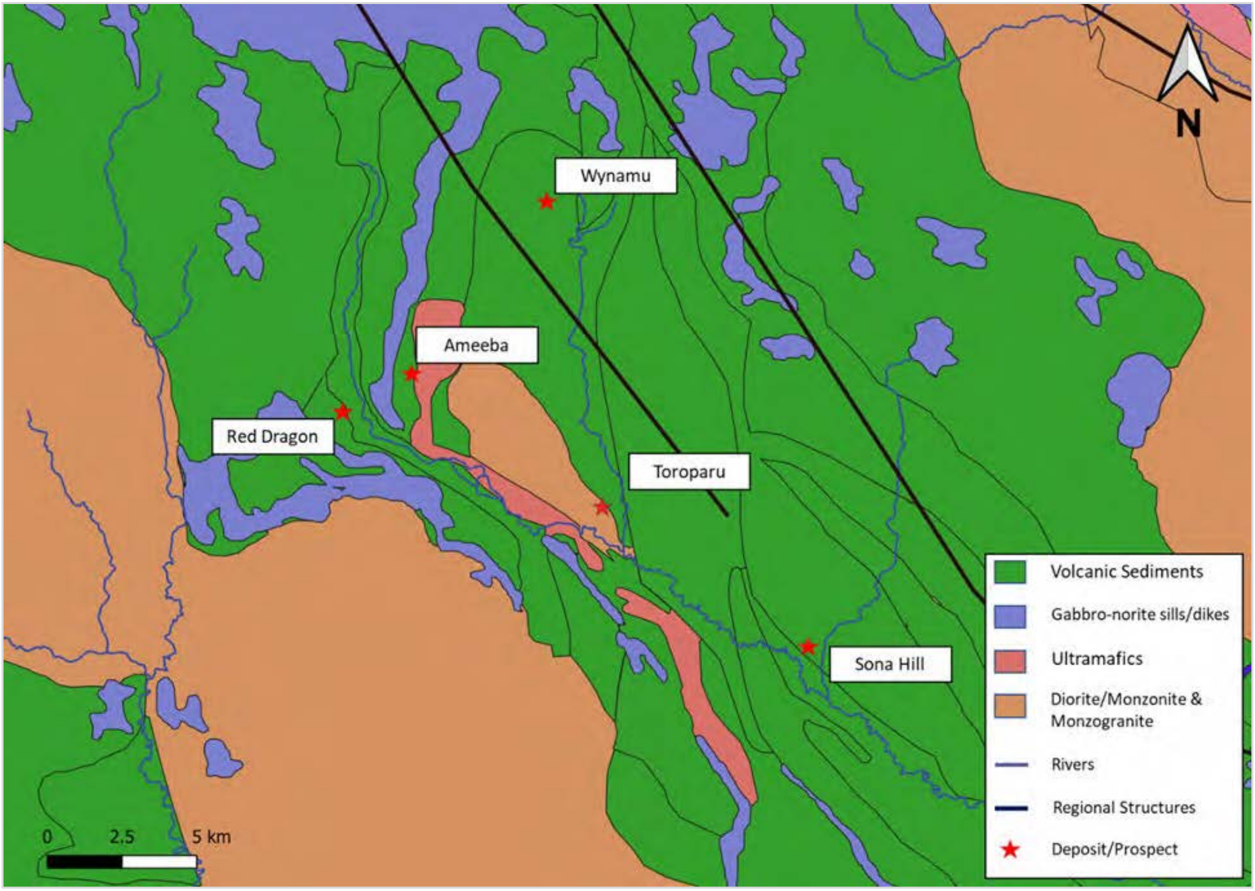


Figure 7-4: Puruni River area geology. Toroparu Deposit show in red located between the Purunia and Wynamu rivers

The regional metamorphic grade is greenschist facies and can reach the amphibolite facies in the vicinity of the granitoid intrusions. Limited lithological information provided by scarce outcrops and exploration drill logs suggests that the belt's central part is predominantly occupied by thick sequences of pyroclastics and sediments, whereas mafic volcanics dominate the border zones. Some strongly weathered rock in road cuts, and associated multi-element geochemistry, suggest the presence of ultramafic facies, which seem to be related to the mafic volcanic sequences.

The Putareng batholith corresponds to a calc-alkaline intrusive complex, ranging in composition from granite and tonalite to diorite. This intrusive complex is thought to be syn- to post-deformational. Exploration revealed the existence of small, more or less elongated, intra-belt plutons, generally of tonalitic to quartz dioritic composition. The Toroparu Deposit developed along the contact zone of one of these small intrusive bodies (Figure 7-3 and Figure 7-4). Reprocessed airborne magnetics data and satellite imagery interpretations provide indications that these small plutons seem to occur preferentially at magnetic low structures along the southern limb of the Puruni VS belt. Several significant gold deposits in Guyana are related to such small intrusive bodies: Aurora, Omai, and Toroparu. Petrographic, geochronological, and litho-geochemical studies are required to investigate in detail the age of the different intrusive phases and their eventual link with gold-(copper) mineralization.

The Upper Puruni Area is marked by sets of NW to WNW and NNW to N-S lineaments.

The NW oriented features seem to constitute typical belt parallel shearing structures, following lithological contact zones, and dominating the regional trend of the belt. The regional structural pattern shows a sigmoidal flexure zone in the northwestern part of the concession, which seems to be controlled by the set of NNW to north-south (NS) lineaments. The flexure zone, if the fractures are strike-slip shear zones, can be an area of right-hand rotational deformation. Unfortunately, there is very little structural information available, making basic and reliable structural analyses difficult.

The Toroparu Deposit is located close to and between two major lineaments (Figure 7-3 and Figure 7-4); the WNW oriented Puruni fault zone, to the southwest, and the NNW striking Wynamu fault, likely affecting the southeast portion of the deposit. Within such a regional structural pattern the mineralized zones of the Toroparu Deposit can be interpreted as east-west oriented, west plunging, dilational zones within an WNW oriented, oblique sinistral strike-slip fault zone. More structural evidence is needed to fully support this interpretation of higher-grade E-W lenses within the overall WNW oriented orebody.

7.2 Property Geology

7.2.1 Toroparu Deposit, Main, and SE Areas

The Toroparu Deposit mineralization is oriented in a west-northwest direction with cross-cutting east-west mineralized structures (Figure 7-5). The system corresponds to a 2.7 km long and 200 m to 40 m wide body and extends to over 400 m in depth. The mineralized body occurs along the northwestern boundary of a tonalitic to quartz dioritic intrusion within a series of mafic volcanics with a thick, gradational layer of saprolite material. Saprolite results from deep tropical weathering, resulting in the larger part of the original rock mineralogy being replaced by clays. Quartz veins and veinlet networks survive quite well in saprolite and contain occasional free gold grains. Sulphides tend to be completely leached and removed, leaving relic voids, and/or oxidized spots. The Toroparu Deposit sits in a topographic low and is near the Puruni and Wynamu rivers. This has resulted in the upper part of the lateritic profile being eroded. Bedrock substratum is overlain by a thin, 1 m residual soil layer, followed by a 10 m 35 m thick saprolite layer. Saprolite rock is the transitional zone between saprolite and fresh rock, creating a gradational contact several metres thick at the Toroparu Deposit.

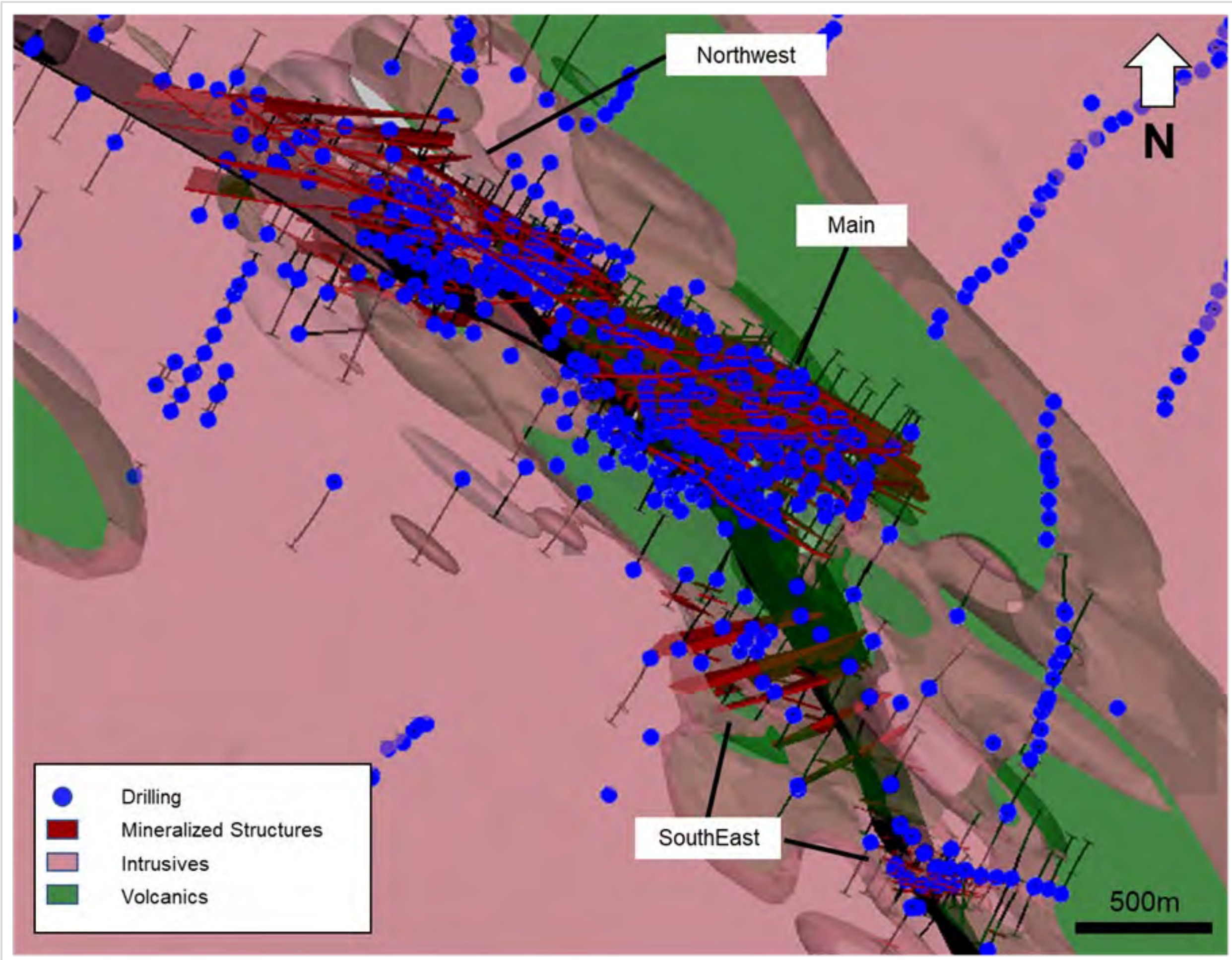


Figure 7-5: Plan view of the geology of the Toroparu Deposit, Main, and SE Areas

The Toroparu Deposit occurs along the northwestern boundary of a tonalitic to quartz dioritic intrusion, close to the southeastern edge of the pluton (Figure 7-5). A series of magmatic intrusives of intermediate mineralogical compositions hosts the mineralization. Detailed core logging shows that the VS sequence and the intrusive rocks did not undergo a strong overall deformation. The volcanics consist of fragmental pyroclastics, alternating with fine-grained tuffaceous layers, coarser grained lapilli, and local intermediate to mafic flows. The VS sequence of alternating coarse and fine volcanoclastics and lava flows appear as massive non foliated layers. Layer limits are generally not well expressed, probably due to strong alteration, making the observations, or interpretations of eventual fold systems difficult. North of the deposit area, the pyroclastics grade into fine-grained and laminated arenaceous and pelitic sediments. At depth, dacite to quartz-andesite has been intercepted.

On a deposit scale, the western part of the Toroparu Deposit mineralization system and the SE area is hosted by intrusive rocks. In the eastern portion, the mineralization forms a cloud along a contact zone of a greenschist metamorphic volcanic sequence, draped over a deeper seated tonalitic intrusive. Intrusive lithologies are tonalite to quartz diorite in composition and display a medium grained granular, massive, but often porphyritic texture. The tonalities and/or quartz-diorites show an overall massive texture. Foliation occurs locally and is probably associated to small local shear fractures. Foliation is relatively frequent in the transition zone between the Main and NW areas and is related to the NW-SE oriented fault system separating the two mineralized bodies. Volcanics and intrusive rocks are intruded by sets of discontinuous sub vertical mafic dykes of variable widths that parallel high-grade NW-SE gold structures (Figure 7-6).

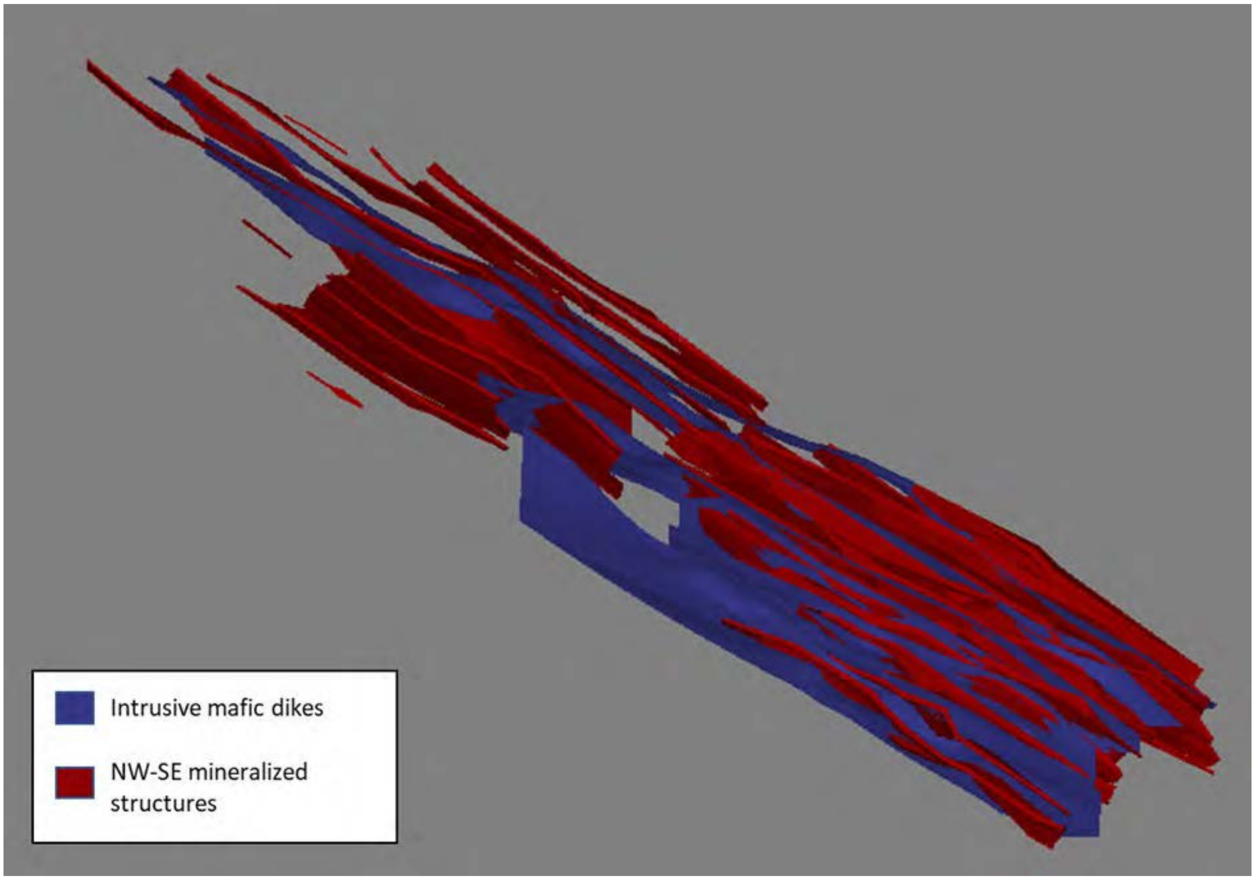


Figure 7-6: Parallel orientation of mafic intrusive dykes to NW-SE mineralized structures within the Toroparu Deposit, Main Area

7.2.1.1 Alteration

The deposit areas are affected by strong hydrothermal alteration. Irregular zones of silicification and sericitization/chloritization, with associated epidote, are pervasive. Carbonate alteration is ubiquitous in most lithologies as small, disseminated grains in the groundmass, sometimes giving the finer grained facies a micro-porphyritic texture. Carbonate minerals are also a common gangue mineral found in quartz veining.

The most common alteration assemblage overprinting rock mineralogy is propylitic/phyllic in nature: albite-actinolite (tremolite)-chlorite – sericite-carbonate-epidote ± local silification. Petrographic analyses describe a hydrothermal assemblage containing secondary alkali feldspars and some rare k-feldspars, suggestive of a transitional propylitic/potassic alteration. Potassic alteration appears to be associated with copper sulphides.

7.2.1.2 Structural Geology

Regional Marawani and Km88 fault structures are tongue-shaped, defined by east-striking lateral ramps that curve into N-striking and W-verging frontal thrust ramps. Local faulting and shearing within the deposit appear as smaller scale lateral ramps with steep to shallow SE plunges. The VS sequence of alternating coarse and fine volcaniclastics, and lava flows appear as massive non foliated layers. Lithology contacts are not well expressed, likely due to strong hydrothermal alteration. Tonalites show an overall massive texture and appear as an undeformed intrusive rock.

Foliation occurs locally and is likely associated with localized shearing. Stronger deformation is frequent between mineralization zones across the deposit and is related to the NW-SE fault system (Figure 7-4 and Figure 7-5). On a deposit scale, relatively dense fracture networks seem to occur by preference in elongated E-W oriented and west plunging lenticular bodies which, in in the Main area, particularly, appear as higher-grade features. Mineralization is associated with roughly E-striking lateral ramps (strike-slip shear zones) and N-striking front thrust ramps. These structures appear to be continuous at depth, where they appear to flatten into a more lystric form. Therefore, they are interpreted to be reverse faults with a left-lateral component, similar to other Archean age gold camps. Shoots of wider and/or higher-grade mineralization in both structures have a steeply dipping SE plunge. A dense fracture network occurs by preference in elongated E-W oriented and west plunging lenticular bodies, which appear to concentrate high-grade mineralization. Due to the strong rheological contrast, dense fracture networks appear to concentrate along or proximal to intrusive-metavolcanic contacts. Fracturing is also seen to cross-cut meta-pyroclastic sequences, hypabyssal intrusives or subvolcanic facies, and tonalite/quartz-diorites.

7.2.2 Sona Hill

Sona Hill Deposit differs from the Toroparu Deposit in the absence of potentially economic quantities of copper mineralization. Gold mineralization is hosted in sub horizontal, shallow dipping structures. It has two sets of identified cross-cutting high-grade gold structures. The Sona Hill property geology is presented in Figure 7-7. The saprolite is generally thicker, as the 25 m to 30 m of topographic hill results in a greater depth to the water table. Sap-rock and saprolite layers can reach up to 60 m thick in the Sona Hill Deposit.

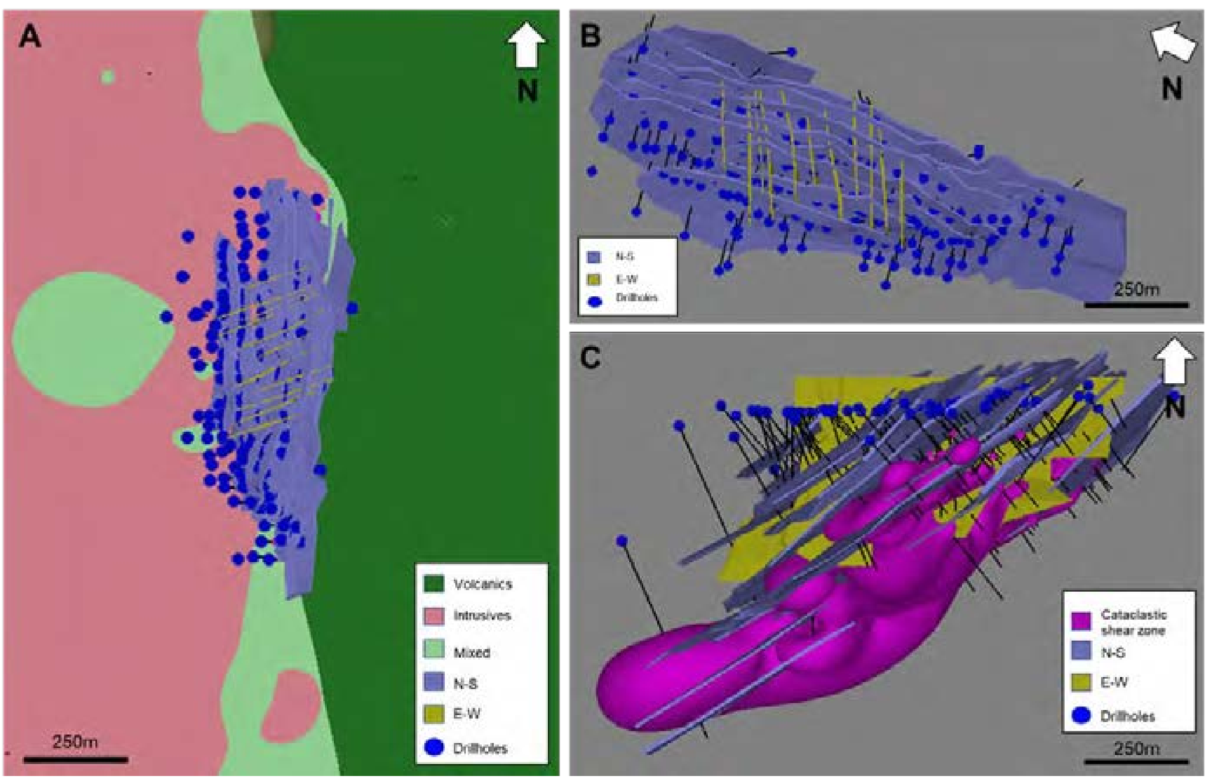


Figure 7-7: Sona Hill Deposit geology. A: Plan view of the Sona Hill Deposit drilling and deposit geology. B: Mineralized structures at Sona Hill Deposit. C: Sona Hill Deposit section view showing hydrothermally altered cataclastic shear zone (pink) which is the main host of mineralization within the deposit.

Local deposit geology determined from drill core consists of intermediate intrusive bodies overlying acidic metavolcanic rocks. The intrusives are affected and underlain by a north trending cataclastic shear zone associated with strong hydrothermal alteration. This zone strikes nearly due north and dips 30º to 40º to the west (Figure 7-7; C).

Similar to the Toroparu Deposit, Main, and SE Areas, mineralization at the Sona Hill Deposit is mainly hosted within intrusive lithologies. These intrusives are petrographically described as porphyritic/micro-porphyritic ± equigranular granodiorite to quartz diorite. Metavolcanics are foliated andesitic volcaniclastics and intermediate to felsic flows. Quartz veining is typically white-crystalline quartz, and can be associated with feldspar, carbonate, tourmaline, sericite, and chlorite with minor sulphides (pyrite). Veins/veinlets are variable in size but generally range from 0.5 cm to 10 cm, density varies significantly. Alteration is quartz-sericite-carbonate-chlorite which is both pervasive throughout the deposit and present as vein halos.

Alteration within the cataclastic shear zone is predominantly phyllic, although locally propylitic/phyllic up to transitional propylitic/potassic assemblages occur. Silica and/or albite ± carbonate flooding result in a banded/layering texture.

The Toroparu and Sona Hill Deposits are a part of a single coherent structural system related to thrusting that carries hanging wall blocks eastward over footwall blocks. Mineralization here is hosted within the frontal part of the back-thrust zone.

7.3 Mineralization

7.3.1 Toroparu Deposit, Main, and SE Areas

Within the Toroparu Deposit, mineralization is hosted by a paleoproterozoic greenschist facies metamorphic VS sequence in contact with a tonalitic to quartz dioritic intrusives. Gold and copper mineralization appears to be largely controlled by a series of moderately developed, dilational brittle-ductile fracture veinlet stockworks. This dilational fracture veinlet stockwork forms a NW-SE trending mineralized corridor with two sets of cross-cutting higher-grade structures (Figure 7-8). Where these structures intersect, there is a large increase in the grade of both gold and copper (Figure 7-10, Figure 7-11, and Figure 7-12). There is also either massive veining or vein breccias in these intersections (Figure 7-8). The Main Area contains the majority of known mineralization, which is still open at depth. The northwestern lens of the main area appears to have slightly lower concentrations of gold grades, but this could be due to a lack of drill density; mineralization here is also open along strike to the NW and at depth.

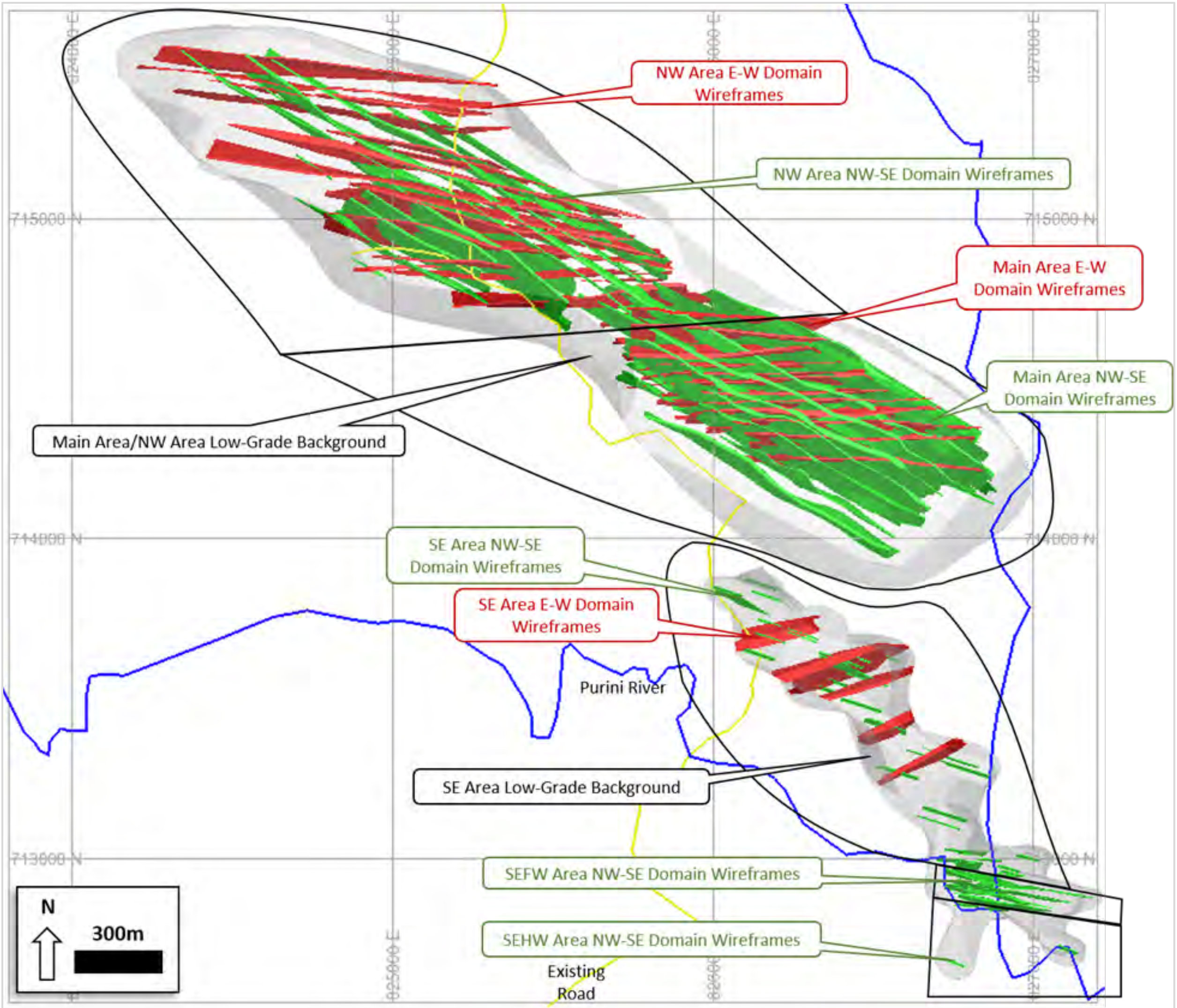


Figure 7-8: Overview of the mineralized domains within the Toroparu Deposit, including the Main, NW, and SE Areas. Green: NW-SE Structures, Red: E-W Structures.

Higher-grade mineralization is concentrated in E-striking within WNW-trending shear zones steeply dipping to the SW. E-striking en echelon zones would be consistent with left-lateral shear sense at the time of mineralization. The left-lateral shearing on either side of the Main and NW areas appears to have created the fracture dense network allowing for the concentration of gold and copper grades. Fracturing is also reduced to intrusive contacts due to the strong rheological contrast. The southeastern mineralization is genetically very similar to the Main area. It is theorized that this is a portion of mineralization offset with the WNW-striking shear zone. Structural measurements taken from oriented diamond drill core can be viewed on the stereonet in Figure 7-9.

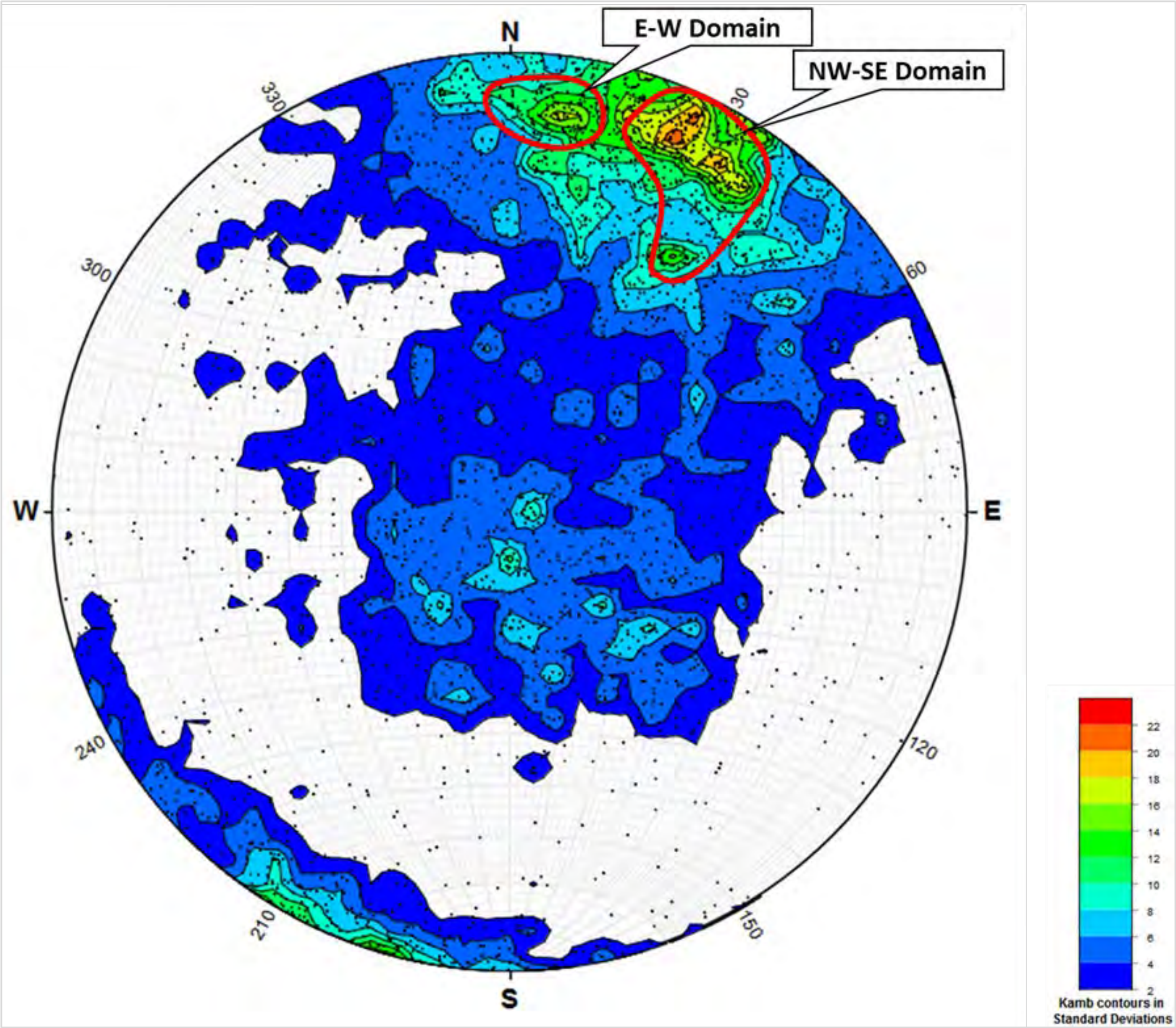


Figure 7-9: Stereonet of measured mineralized structures within the Toroparu Deposit

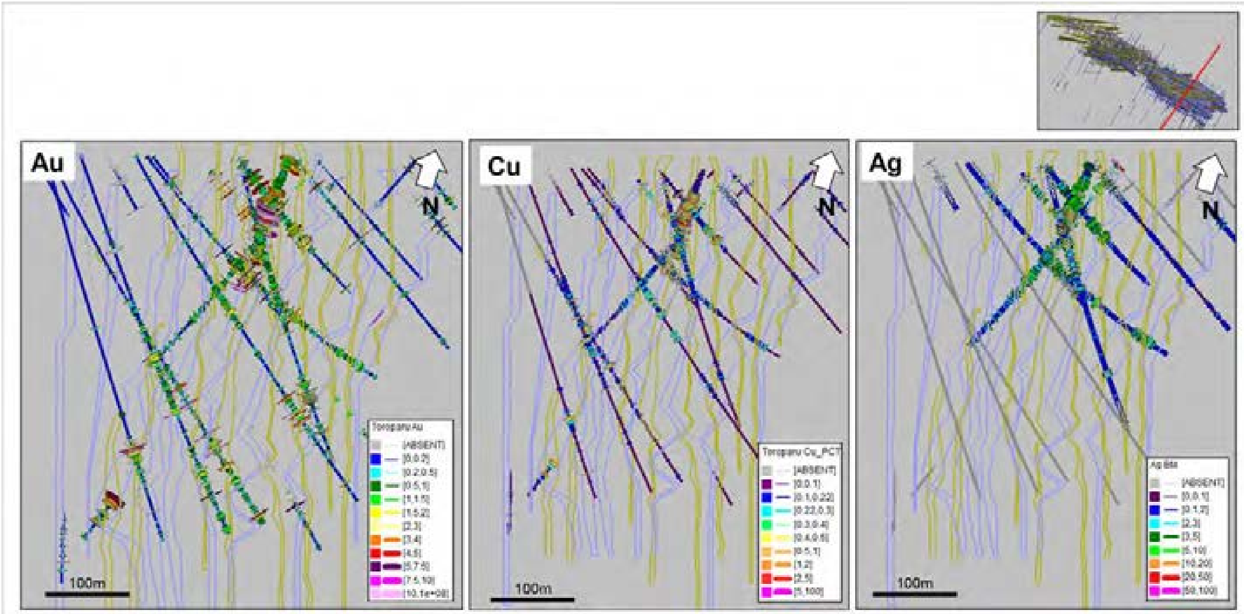


Figure 7-10: Cross-section of mineralization within the Toroparu Deposit, Main Area. Purple wireframes are NW-SE and yellow wireframes are E-W directed.

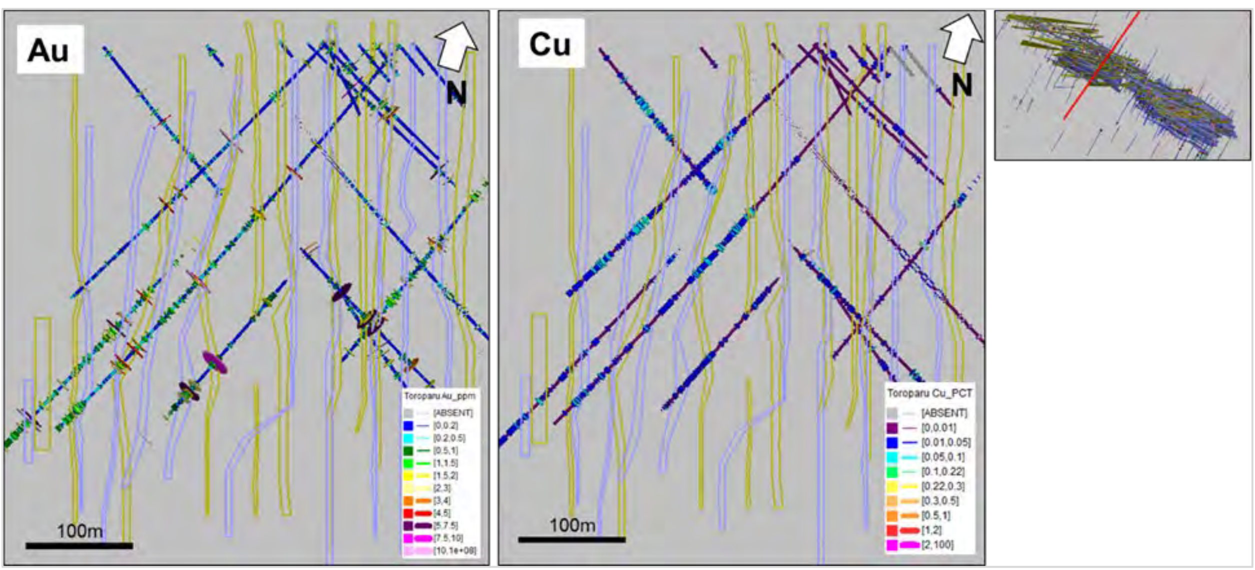


Figure 7-11: Cross-section of mineralization in the Toroparu Deposit, NW Area. Purple wireframes are NW-SE and yellow wireframes are E-W directed.

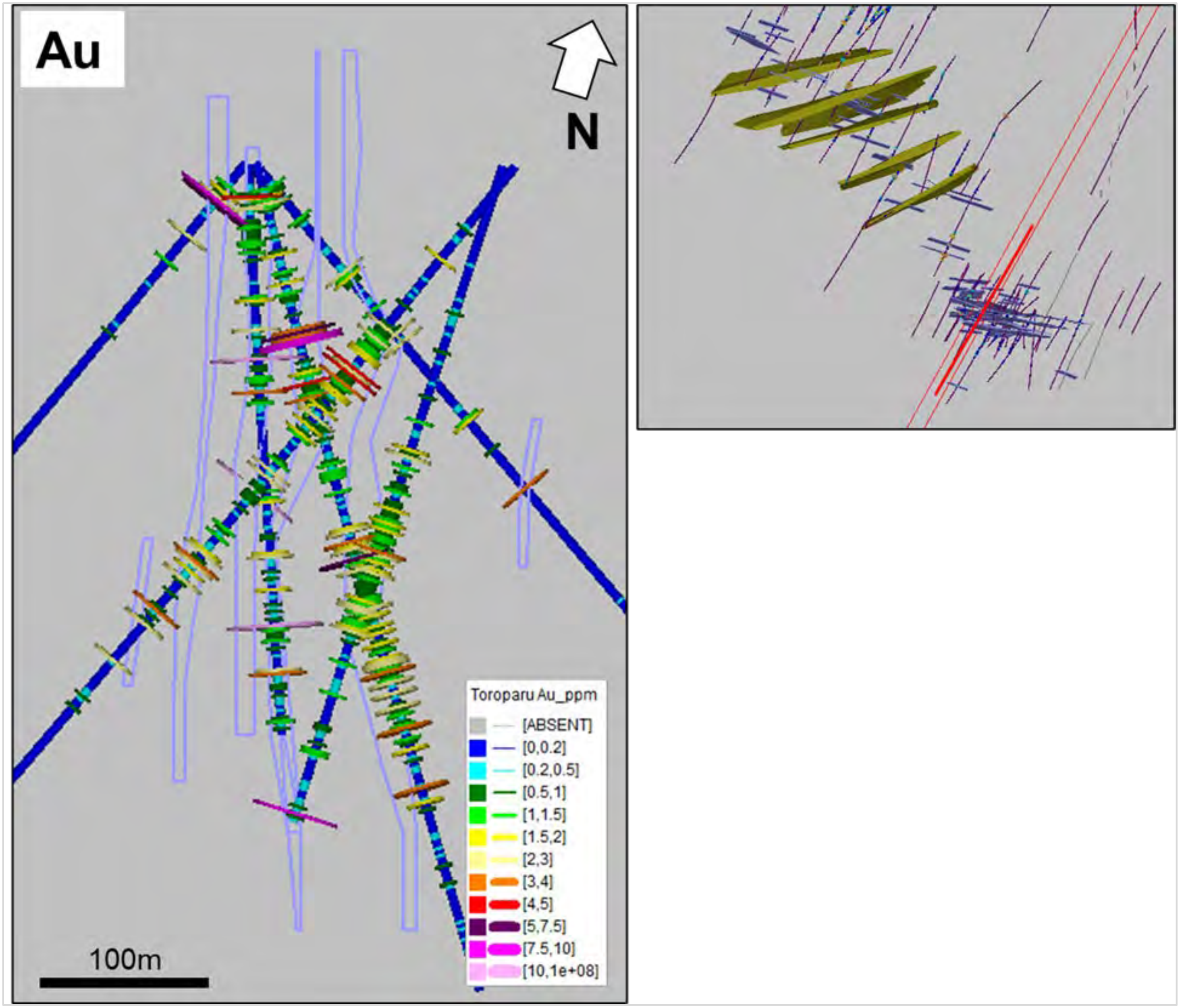


Figure 7-12: Cross-section of mineralization at the SE Area of the Toroparu Deposit

Gold mineralization in the saprolite and the low-grade fresh rock are similar with no apparent depletion. For this reason, saprolite grades are included when estimating the background grades.

Mineralization at the Toroparu Deposit is associated with quartz-carbonate veinlets with minimal sulphides (typically 0.5% to 1%, a maximum of 3%). Core logging indicates that the primary style of gold-copper mineralization is fine to coarse grained disseminations of sulphide blebs, aggregates, and clusters of chalcopyrite and subordinate pyrite, bornite, molybdenite, chalcocite, and very rarely arsenopyrite. Pyrite is less widespread but seems to occur in a large halo around the gold-copper mineralized zones. Sulphides can be disseminated in the rocks, but predominantly occurs in the fine quartz-carbonate veinlets and/or fractures, which seem to control the mineralization. Zones of higher-grade gold-copper mineralization are associated with the presence of higher concentrations of bornite and somewhat more abundant molybdenite, and denser fracture/veinlet networks. Furthermore, the quartz-carbonate veinlets, and fine fracture networks contain frequent visible fine gold grains. Copper mineralization disappears or becomes very weak in the western mineralized lens, where intermediate porphyritic intrusives are predominant, and fracture networks are less well developed.

Inductively coupled plasma mass spectrometry (ICP-MS) analyses were carried out on samples from a selected number of core holes. The results indicate that samples within the main mineralized zones are anomalous in gold, copper, silver, bismuth, tellurium, selenium, tungsten, tin, lead, and molybdenum. Within the gold-copper higher-grade zones, silver correlates well with copper. In addition, it often reaches significant concentrations, such that it can be considered as a by-product. This is confirmed by recent metallurgical testing.

The mineralogical and limited chemical information and the presence of a porphyritic intrusive of intermediate composition suggest that the Toroparu Deposit belongs to the class of porphyry gold-copper deposits. On the other hand, fracturing patterns suggest mineralization results from dilational mechanisms related to oblique strike-slip shearing, creating a more orogenic lode gold style of deposit. It is possible that this deposit formed as a gold-copper porphyry style mineralization early in the geological evolution. The possible shear zone related (orogenic) signature of the deposit can be explained by eventual partial remobilization during a later period of shearing in a regional compressional phase. Logging of diamond drill holes drilled parallel to higher-grade zones (>1.5 g/t) and results of the borehole scanning survey reveal a predominant east-west fracture set. On a deposit scale relatively dense fracture networks seem to occur by preference in elongated E-W oriented and west plunging lenticular bodies, which, in particular in the Main and SE Areas, appear as higher-grade features. A similar feature, but less well expressed because of lower grades, has been detected in the western part of the deposit.

7.3.2 Sona Hill Deposit

The Sona Hill Deposit Mineral Resource is principally associated with shallowly W dipping quartz veinlets within an intrusive body about its W dipping fault contact with underlying volcanics (Figure 7-13). Drill core logging observations suggest that the Sona Hill shear zone is poly phased. Economic veining is concentrated within hydrothermally altered intrusives. In contrast to the Toroparu Deposit, mineralization in the Sona Hill Deposit consists of gold within quartz-sericite-carbonate-chlorite alteration.

Gold-pyrite mineralization is hosted by an irregular network of dispersed quartz-tourmaline-feldspar veins within intermediate intrusives surrounded by halos of intense bleaching (hydrothermal alt) and abundant finely disseminated magnetite. Mineralization occurs at or above the contacts of a cataclastic hydrothermally altered shear zone. The quartz veins appear parallel to subparallel with local foliation. Associated sulphides are minor, estimated at 1% to 3%, and consist only of pyrite. Other associated gangue minerals include feldspar, carbonate, tourmaline, sericite, and chlorite. There is also only one set of high-grade structures identified at the Sona Hill Deposit. Occasionally, small pockets of gold occur in underlying volcanics. Cross-sections of high-grade mineralization as a result of intersecting structures are displayed in Figure 7-15.

Figure 7-13: Overview of the Sona Hill Deposit mineralized structures. Red: East-West oriented QZO structures. Green: NS oriented structures.

Structural logging data and the distribution and frequency stereonet plots demonstrate that the folded quartz-tourmaline veins appear to constitute a pattern of two veins sets:

- NS oriented/W dipping, foliation subparallel veins – shear and oblique
- E-W oriented/S dipping veins – generally larger, likely extensional

Vein orientations vary from N-S over NW to W-E, but a majority are more or less parallel to the orientation of the underlying shear zone and seem to form more or less subparallel sheets of strong alteration and veining (hydro fracture zones) above and at the contact of the shear zone (Figure 7-14 and Figure 7-15). The larger veins (> 20 cm) generally provide the best gold grades. The gold grade is generally proportional to the degree of abundance of pyrite. Structural measurements taken from the core led to identifying quartz veins that were 30^{o} to the core axis. These were later identified to be a set of E-W oriented mineralized structures (Figure 7-13 and Figure 7-15).

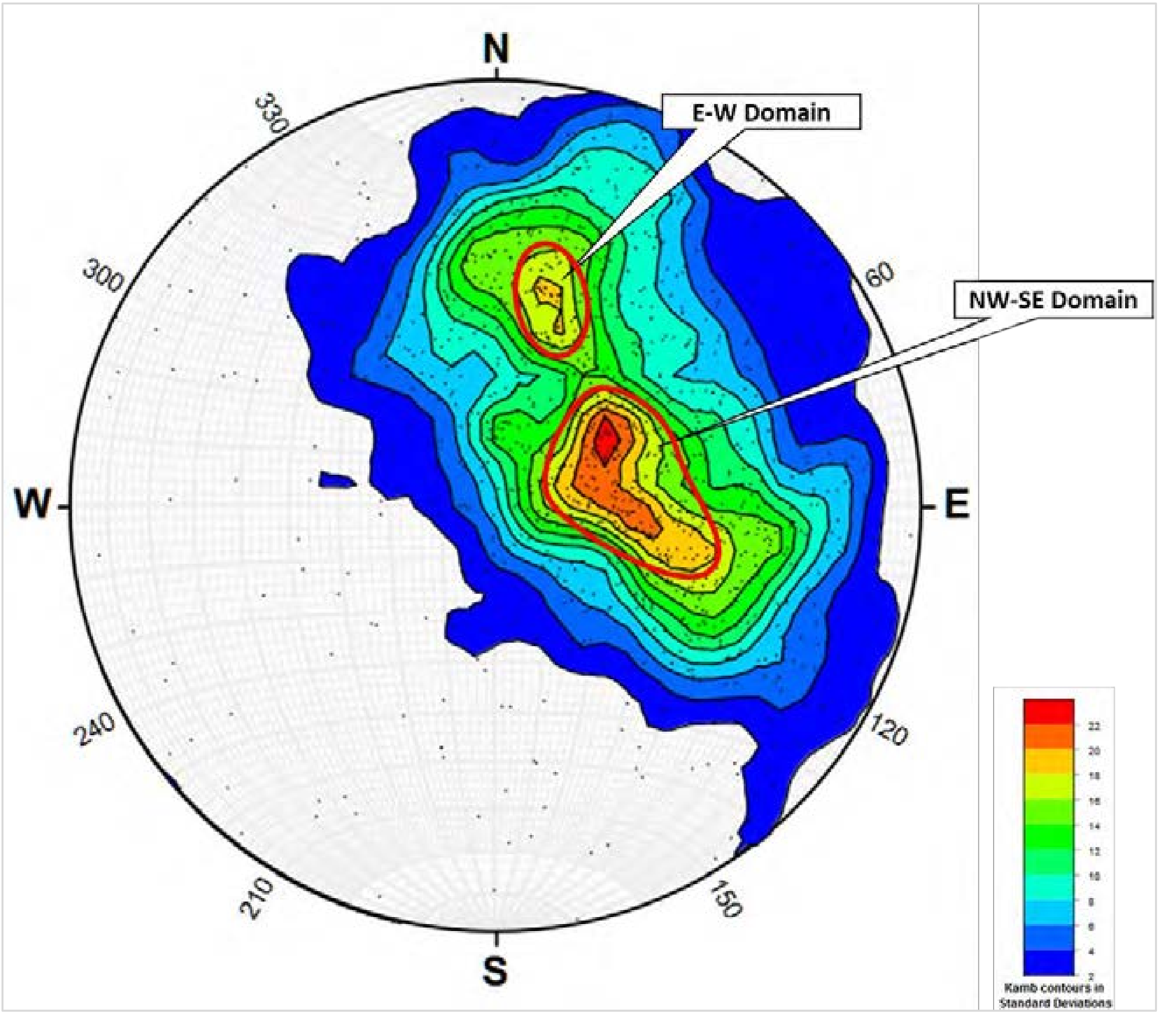


Figure 7-14: Stereonet created from structural oriented drill core measurements at the Sona Hill Deposit. Two sets of distinct structures are identified.

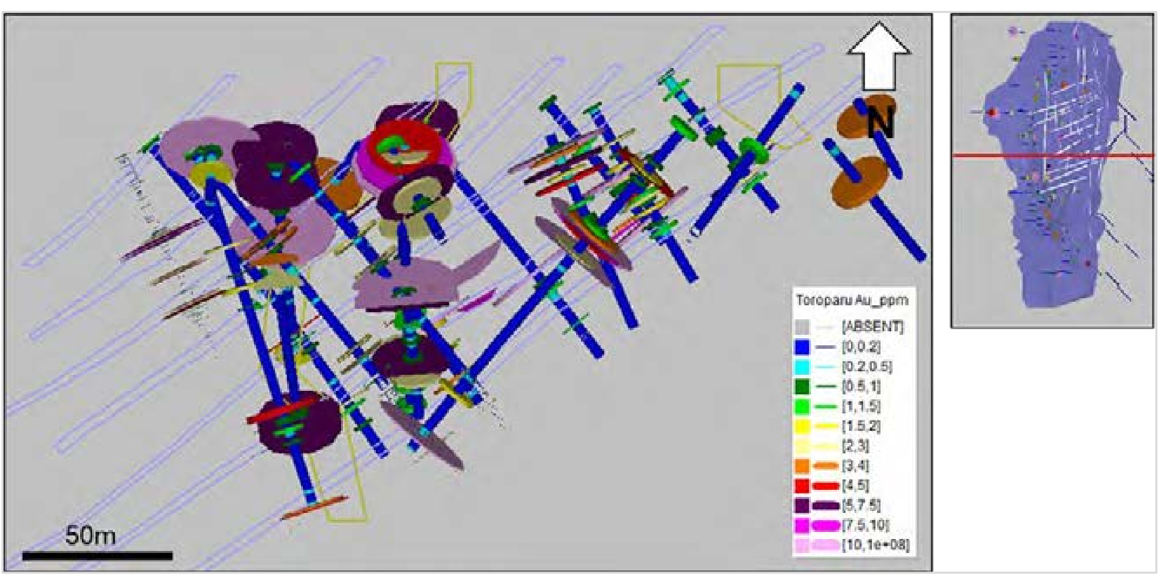


Figure 7-15: Cross-section of mineralization at the Sona Hill Deposit.

8 DEPOSIT TYPES

The Toroparu Deposit, Main, and South East areas are representative of an Archean gold (±copper) intrusion related system (Kontak, Katz, & Dubé, 2012) (Katz, Kontak, Dubé, & McNicoll, 2015) (Mathieu, Crépon, & Kontak, 2020). It closely resembles other structurally controlled, mesothermal gold systems seen in Archean greenstone belts such as the Abitibi or the Yilgarn craton. It appears genetically similar to deposits such as Côté Gold, Lac Doré, Boddington and Las Cristinas (Table 8-1). Deposits of this type are closely associated with intrusive suites with geochemical signatures of rocks derived from hybrid melts (high heavy rare earth element, abundance of intermediate units, low- to moderate water content) (Mathieu, Crépon, & Kontak, 2020). They include porphyry copper-gold, syenite-associated disseminated gold, and reduced gold-bismuth-tellurium-tungsten intrusion related deposits, as well as stockwork disseminated gold. It is thought that this could be a result of the overprinting of two different deposit types. The Sona Hill Deposit represents an orogenic lode gold deposit and is related to the gold ± copper system at the Toroparu Deposit.

Table 8-1: Toroparu Deposit Style Versus Other Deposits with Similar Mineralization from the Guiana Shield, Yilgarn Craton, and the Abitibi Subprovince

Company: **Country:** **Commodity:**	**Toroparu** **Guyana** **Au-Cu-(Ag)**	**Christinas** **Venezuela** **Au-Cu-Ag**	**Aurora** **Guyana** **Au**	**Côté Gold** **Canada** **Au-(Cu)**	**Copper Rand/Cedar Bay** **Canada** **Au-Cu**	**Boddington** **Australia** **Au-Cu**
Host Lithologies Associated Rocks	Metavolcanics, intermediate-mafic tonalite intrusive complex, Q-diorites	mafic volcanics, minor sediments	Tonalite-diorite, sediments, mafic volcanics	Brecciated and altered tonalite and diorite, synite intrusives, structurally hosted orogenic lode gold in metavolcanics	Meta-Anorthosite, grabbro, tonalite, diorite, pyroxenite	intermediate-felsic intrusives, volcanics, diorite, andesite, and dacite
Metamorphic Grade	Lower Greenschist	middle greenschist	lower greenschist	lower greenschist	lower greenschist	greenschist – lower amphibolite
Structural Setting	WNW shear zones, brittle fracture network	brittle-ductile shearing	vein stockwork (intrusive), brittle-ductile shearing in volcanic-sediments	Brecciated stockwork veining proximal to second and third splays off crustal scale regional faults	Brittle-ductile shears/faults, secondary extensional fault off of crustal scale regional fault, anticline	Shear Zones, brittle-ductile faulting, reactivated veins/veinlets/lenses
Ore/Gangue Minerals	chalcopyrite-bornite-pyrite-molybdenite-chalcopyrite-chalcocite, quartz, carbonate	pyrite, chalcopyrite, covellite, molybdenite, quartz-carbonate-tourmaline	pyrite, quartz, ankerite	pyrite, chalcopyrite, pyrrhotite, magnetite, molybdenite	pyrite-chalcopyrite-pyrrhotite-sphalerite-galena, quartz, carbonate	chalcopyrite, pryite, pyrrhotite, molybdenite, tellerium, bismusth, molybdenite, tungsten
Hydrothermal Alt	silica, sericite, chlorite, carbonate, sulfidation, albitization	tourminalization, sericite, chlorite, silica, biotite	silica, sericite, albite, carbonate, sulfidation	biotite, chlorite, sericite, silica, sodic	sericite, silica, albite, epidote	silica, albite, sericite, biotite, actinolite, chlorite, epidote
Structural Timing of Mineralization	late to post-tectonic	syn- to late-tectonic	syn- to late-tectonic	syn-intrusive	late- to post-tectonic	late-tectonic

8.1 Gold Porphyry Deposits

Porphyry deposits occur worldwide as a series of extensive, relatively narrow, linear metallogenic provinces. They are predominantly Mesozoic to Cenozoic in age and are associated with orogenic belts, specifically convergent plate boundaries. This deposit type is typically large, containing hundreds of millions of tonnes of ore. Grades vary considerably but generally average less than 1%. Porphyry deposits with average gold contents of ≥0.4 g/metric ton (t) gold may be defined as gold-rich (Sillitoe, Some thoughts on gold-rich porphyry copper deposits., 1979). Although, a few deposits are richer in gold contents containing several hundred million metric tons (Mt) averaging >1.5 g/t gold (Sillitoe, 2000). Typically, gold-rich porphyry deposits are also deficient of molybdenum (Sillitoe, 2000).

Many gold-rich copper porphyries consist of both pre and post mineralization intrusives. Mineralization is typically associated with intrusions belonging to exclusively I-type magnetite suites (Sillitoe, Gold-rich porphyry deposits: descriptive and genetic models and their role in exploration and discovery., 2000). These intrusions are highly oxidized, sulphur-poor representatives of the magnetite series. The porphyry stocks span a range of compositions from k calc-alkaline diorite, quartz diorite, and tonalite through to high K calc-alkaline quartz monzonite to alkaline monzonite and syenite. Pre-mineralization textures are generally equigranular and related to porphyry stocks. Post mineralization intrusives typically include dykes, plugs, and diatremes. Hydrothermal breccias occur as early orthomagmatic and/or phreatic and phreatomagmatic varieties.

Gold mineralization at gold-rich porphyry deposits is introduced during K silicate alteration and contents of both gold and copper will vary greatly (Sillitoe, 2000). Gold contents correlate strongly with vitreous quartz-magnetite-chalcopyrite veining, often accompanied by a K-feldspar halo. Ore zones are normally upright cylinders or bell-shaped bodies. Gold contents tend to increase, even double, at depth over several hundred metres (MacDonald & Arnold, 1994) (Sillitoe, 2000). Gold is fine-grained, <20 µm, generally <100 µm and is closely associated with iron and copper-iron sulphides (pyrite, bornite, chalcopyrite).

Common alteration types of gold-rich porphyries include:

- Ca-Na silicate alteration
- K silicate (potassic) alteration
- Propylitic alteration
- Intermediate argillic (sericite-clay-chlorite)
- Sericitic
- Advanced argillic

The Côté Lake gold deposit, Chibougamou-type copper-gold deposits and the Las Cristinas and Las Brisas deposits of Venezuela (150 km west of the Toroparu Deposit) fall within this category.

8.2 Shear Zone Hosted Deposits

Lode gold deposits are a characteristic feature of late-Archean granitoid-greenstone terranes and are classified as “mesothermal.” They typically occur in environments ranging in metamorphic grade from greenschist to lower amphibolite facies. Several authors have shown that these deposits formed during the late stages of evolution of the host terrane (Gebre-Mariam, Hagemann, & Groves, 1995) (Kerrick & Woodsworth, 1989).

The main host lithology tends to be volcanics of theoliitic affitnites and strong spatial and temporal relationship with granitoids of varying compositions, typically tonalite- and granodiorite-dominated large volume batholiths (Mathieu, Crépon, & Kontak, 2020). Within the Abitibi greenstone belt it has been observed that copper rich gold deposits show an association with large volume batholiths containing diorite (Katz, Kontak, Dubé, & McNicoll, 2017) (Mathieu, 2019) (Mathieu, Crépon, & Kontak, 2020). Common alteration types associated with mineralization include sericitization, silicification, albitization, strong chlorite alteration, and biotite.

Gold deposition occurs adjacent to first-order, deep-crustal scale fault zones. Typically, these faults are reverse and listric at depth. These faults can be several hundred kilometres, such as the Porcupine-Destor or Cadillac-Larder-Lake deformation zones, and show complex defomational, and structural histories. Economic mineralization occurs within second and third order structures. Styles can vary from being vein hosted within brittle-ductile zones, stockworks, and breccias, as well as replacement and disseminated-type orebodies in deeper ductile environments.

Mineralization can be disseminated, or vein hosted and displays a timing that is structurally late and is syn- to post- peak metamorphism. Associated sulphide minerals include pyrite, pyrrhotite, chalcopyrite, and arsenopyrite.

The Boddington gold-(copper) deposit within the Yilgarn craton of Australia is a good example of an Archean greenstone hosted lode gold deposit that has overprinted a copper porphyry deposit. It exhibits similar mineralization to that found in the Toroparu Deposit. The Côté Gold Deposit is an Archean low-grade, high tonnage gold (± copper) discovery with a similar mineralization style to the Toroparu Deposit. This type of deposit shows a strong spatial association to quartz dioritic intrusives and major regional deformation zones.

8.3 Comments on Deposit Types

Understanding deposit types is important to guiding continuing exploration activities. Due to the lack of in depth exploration throughout the Guiana Shield, compared to other Archean greenstone belts such as the Abitibi and Yilgarn Craton, not as much detail is known of the deposit types hosted within. Similar to the Boddington deposit, the Toroparu Deposit is hosted by greenschist facies volcanics, sub volcanics, and intrusives. They are both structurally controlled and display overprinting of two hydrothermal systems.

- Gold is closely associated with zones of complex quartz-carb veining and increased alteration.
- Hosted within or at the contact of diorite – tonalitic intrusive complexes.
- Strong association with regional crustal scale reverse listric faults.

Compared to Chibougamou-type copper-gold deposits of the Abitibi province in Canada, there are several similarities. Firstly, sericitization alteration halos, irregular networks, sometimes brecciated, quartz-carbonate veins are key characteristics of mineralized areas. However, mineralization in this deposit type also correlates with semi-massive to massive sulphide mineralization typically containing pyrite-chalcopyrite-pyrrhotite-sphalerite and galena. They are also mainly hosted within intrusive bodies, similar in composition to those found at the Toroparu Deposit.

9 EXPLORATION

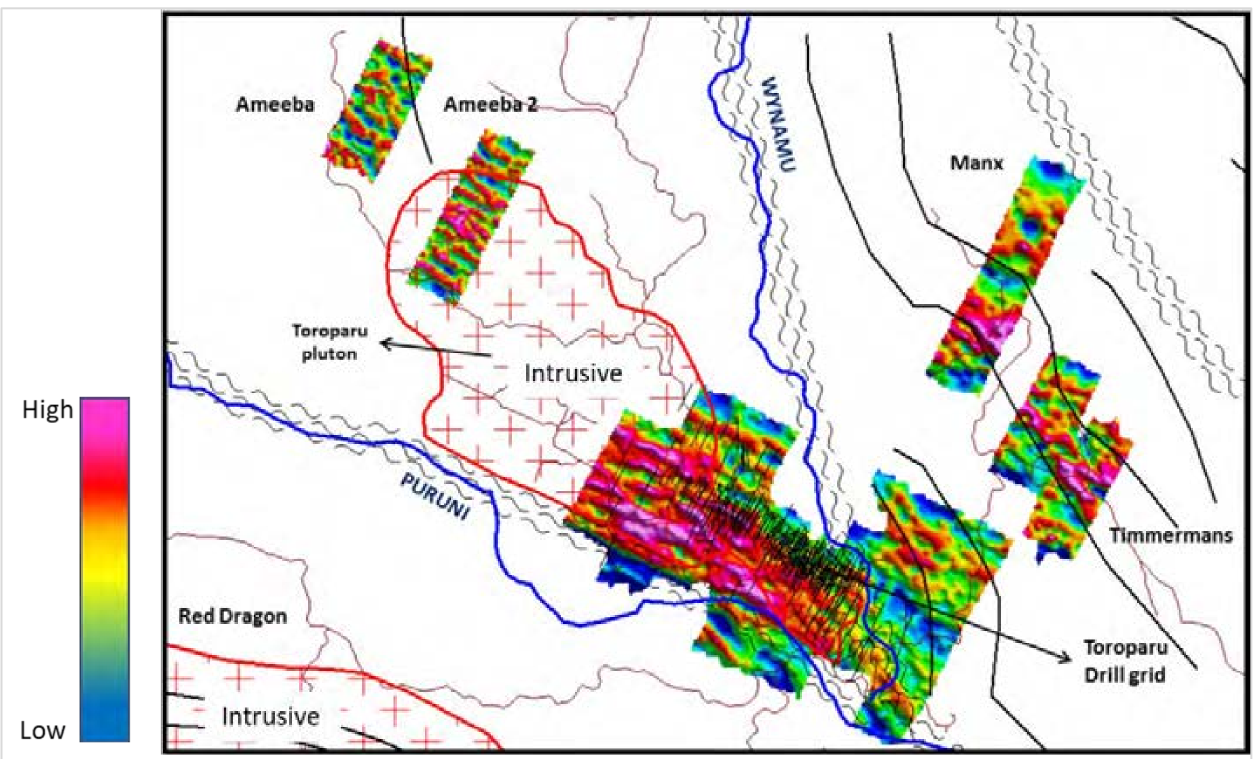
9.1 Grids and Surveys

Systematic grid sampling as regional exploration is a serious challenge in Guyana's dense inaccessible rainforest. A regional grid 1,000 m x 1,000 m was applied, covering a total surface of 275 km^2. Tracks were created using Hägglund vehicles on tracks. Later sampling grids were built off this original grid.

9.2 Exploration

Due to tropical rain forest conditions reflecting intensive deep weathering, the Project area and surroundings are poorly outcropped. Furthermore, the Toroparu Deposit is in the Puruni Valley, constituting the lowest topographic level of the area. Consequently, rock/saprolite outcropping is very poor, which renders geological mapping particularly difficult. Various geochemical, geophysics (IP and magnetic surveys) (Source: Sandspring's 2011 Annual Exploration Report

Figure 9-1), reverse circulation (RC) drilling and diamond drilling campaigns have taken place across the Property and associated prospects (Table 9-1 and Table 9-2).



Source: Sandspring's 2011 Annual Exploration Report

Figure 9-1: Location of the ground magnetic survey geophysics grids – 2010

Table 9-1: Exploration Campaigns Across the Toroparu Deposit Including Geophysical, Geochemical, RC, and Diamond Drilling Surveys

Year	Company	Area	Activity	Comment
Dec 2006 – Dec 2009	ETK Inc.	Toroparu	Diamond Drilling	Initial diamond drilling across the Toroparu Deposit.
2010	Sandspring Ltd.	Upper Puruni	Geophysics	Ground geophysics program completed by Insight Geophysics. Combined Magnetometer and Gradient Array IP along 200 m spaced lines in an area of 4,800 m x 2,800 m. IP resistivity shows consistent anomalies corresponding to Toroparu granodiorite pluton. Chargeability was low over areas of high gold-copper mineralization despite sulphide presence.
2011	Sandspring Ltd.	Toroparu Main	Diamond Drilling – Resource Infill	120 holes for a total of 45,549 m to increase overall resources and average gold-copper grades. Identified main mineralization host as Paleoproterozoic VS sequence. Fracture zone controlling grade, quartz-carb vein stockwork
	Sandspring Ltd.	Upper Puruni	Surface Sampling Mapping	Surface sampling/mapping in surrounding NW, N, and NE areas of Toroparu including Ameeba, Manx, and Timmermans alluvial prospects. Puruni Valley which is thought to be an expression of regional NW trending shear. Centre west zone of concession package (also known as red dragon) showed structure and high magnetic signatures. 2,891 samples were collected.
2012	Sandspring Ltd.	Toroparu Main	Diamond Drilling	Updated and infill resource for Resource model Update at the beginning of 2013. Program consisted of 34,055 m and 142 holes.
	Sandspring Ltd.	Upper Puruni	RC drilling	Testing gold anomalies revealed by saprolite Geochem program from 2011. Ameeba results justified limited further test drilling. The Red Dragon Zones revealed no significant results.

Year	Company	Area	Activity	Comment
2013	Sandspring Ltd.	Makapa	Geochemical sampling	378 saprolite samples were collected for multi-element analysis including gold, the final grid was 200x100m. Geochem results confirmed gold anomaly, NNE oriented, 1 km long 500 m wide feature of continuous 100+ ppb values, covering two small hills. A dozen samples reach high values of >500 ppb. Resulted in recommended RC drilling follow up work.
2015	Sandspring Ltd.	Otomung River	Geochemical Sampling	951 samples were submitted to Bureau Veritas Lab in Vancouver. Only weak gold values were seen, but a clear 8 km trend similar to the Toroparu Deposit was identified along a previously interpreted intrusive body.
2016	Sandspring Ltd.	Wynamu Hill	DDH	A gold anomaly was discovered on the property by geochemical surveys conducted in 2012 and 2013. 1,127 m of drilling and 14 shallow holes were allotted to further exploring the anomaly. Some significant gold intercepts, 1.18 g/t over 19.5 m and 7.51 g/t over 21.5 m, justify further exploration in the area in the future.
	Sandspring Ltd.	Otomung River	Geochemical Sampling	660 samples were collected on a 250x100m grid over gold anomalies. Additional 305 samples were collected from 30 km^2 area extending the 2015 grid. Allowed for distinguishing of 8 km long gold-molybdenum feature interpreted to be an elongated pluton. Justified further infill sampling.
2017	Sandspring Ltd.	Otomung Saprolite	Geochemical Sampling	Grid was tightened over anomalous zone discovered in 2016 geochemical sampling. New grid was 100x125m and 50x125m in the northern anomaly. A total of 885 samples were collected.

Year	Company	Area	Activity	Comment
2018	Sandspring Ltd.	Wynamu Hill	DDH	5,300 m of drilling in 48 holes. Several holes intercepted gold mineralization in saprolite and fresh rock.
	Sandspring Ltd.	Ameeba	DDH	Since Sona Hill discovery, the Ameeba geological context has been re-considered. 2018 drilling was focused on finding gold resources in saprolite which is thick at Ameeba. Concluded that gold intercepts here are narrow and scattered.
2019-2020	Gold X	Toroparu Main	DDH	In 2020 Nordmin Engineering Ltd. re-evaluated the resource model for Toroparu Main and SE and identified possible sets of high-grade cross-cutting structures. Diamond drilling was planned to confirm the presence of these structures and to increase drill density. Significant intercepts containing visible gold (VG) confirmed the presence of cross-cutting structures.

Table 9-2: Exploration Campaigns Across the Sona Hill Deposit

Year		Area	Activity	Comment
2012	Sandspring Ltd.	Sona Hill	RC/Diamond Drilling	RC drilling showed encouraging significant gold intercepts – this resulted in a small exploration DDH program. Results demonstrated scattered gold mineralization with no significant copper results. The distribution of grade appeared irregular.
2015-2016	Sandspring Ltd.	Sona Hill	DDH	8,084 m were drilled in 68 holes along E-W oriented drill fences and one NS drill section. Avg drill density 50 m x 50 m.
	Sandspring Ltd.		Geochemical Exploration	Geochemical surveying was run concurrently with geophysical survey program, which filled in to a tighter 100 m x 50 m grid over the same area to add data. Geochemical data combined with geophysical data lead to positive results and indicated a need for further exploration.

Year		Area	Activity	Comment
	Sandspring Ltd.		Ground Geophysics	18 line-km IP and magnetics ground geophysical survey was conducted over saprolite geochemical survey grid established in 2012. Results suggested an extension of the west dipping low angle, strongly altered, shear zone to the west. Indicated potential for more mineralization in its hanging wall. These results combined with Geochem indicate further exploration and potential to increase resource inventory.
2017-2018	Sandspring Ltd.	Sona Hill	DDH	6,8975.5 m in 58 holes along E-W oriented fences and a second N-S drill section. Average drill density pattern of 25 m x 25 m. The aim was to convert Inferred into Measured and Indicated categories.

9.2.1 2010-2011 Surface Exploration

Until the beginning of 2011, the Upper Puruni Concession package (1,000 km^2) had remained unexplored. A systematic surface sampling and mapping approach was implemented starting in 2011, focused primarily on geological potential for gold and/or base metals. Targets were originally selected from interpretations of airborne geophysical data and sat imagery (Source: Sandspring's 2011 Annual Exploration Report

Figure 9-1). In areas where geochemical sampling yielded positive results, tighter grid spacing for ground geophysics was carried out.

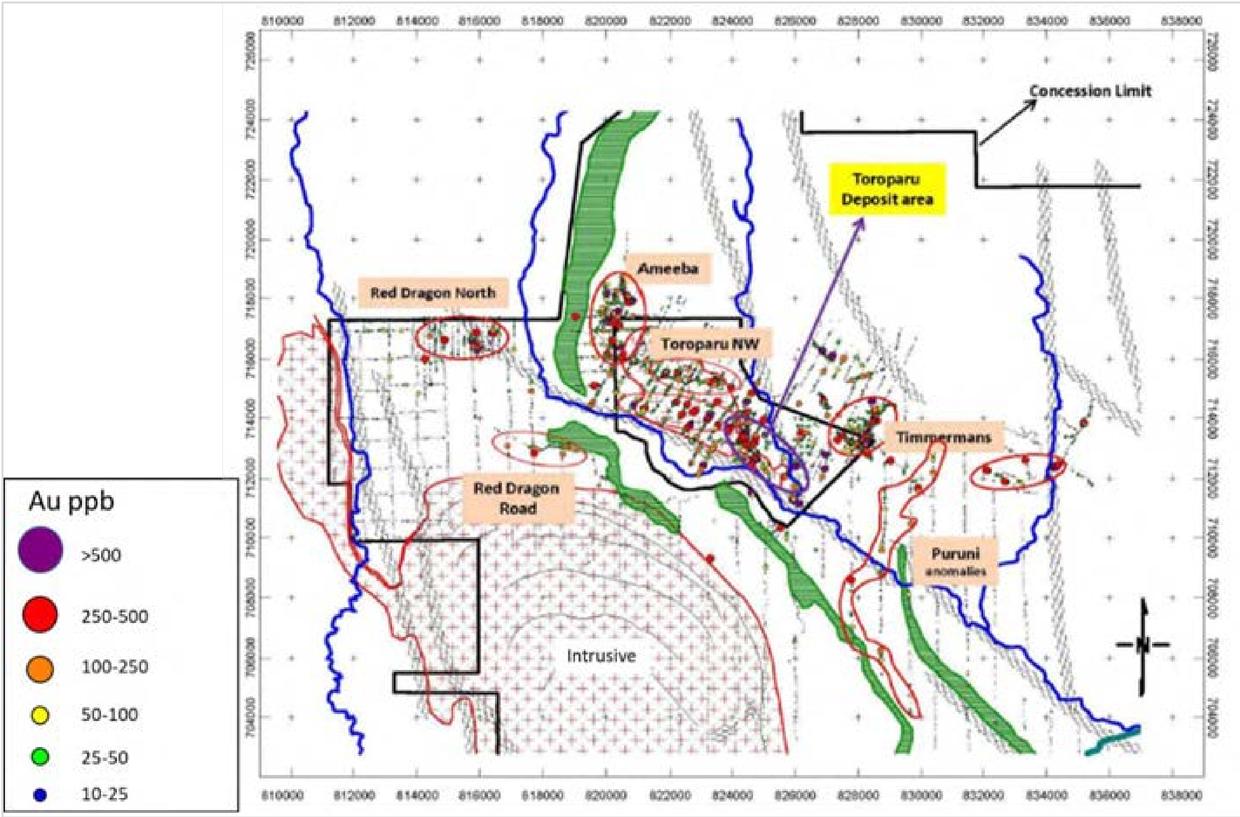
Geochemical samples were taken from the soil layer, and if possible, the laterite layer, averaging 0.5 m to 0.3 m depth. This was done using a hand auger. QA/QC samples were inserted (blank, standard, field duplicates) at intervals of 20 samples. Samples were dried and prepped in the on site ACME Prep lab and submitted for an ICP 1F03 analysis (30 g sample – 37 multi-element analysis). A total of 4,389 samples were collected (Source: Sandspring's 2011 Annual Exploration Report

Figure 9-2).

The geochemical sampling resulted in identifying the Toroparu Deposit NW area, the Ameeba hills geochemical anomaly – which led to follow up DDH drilling – a possible extension of the Toroparu Deposit (Source: Sandspring's 2011 Annual Exploration Report

Figure 9-1 and Source: Sandspring's 2011 Annual Exploration Report

Figure 9-2).



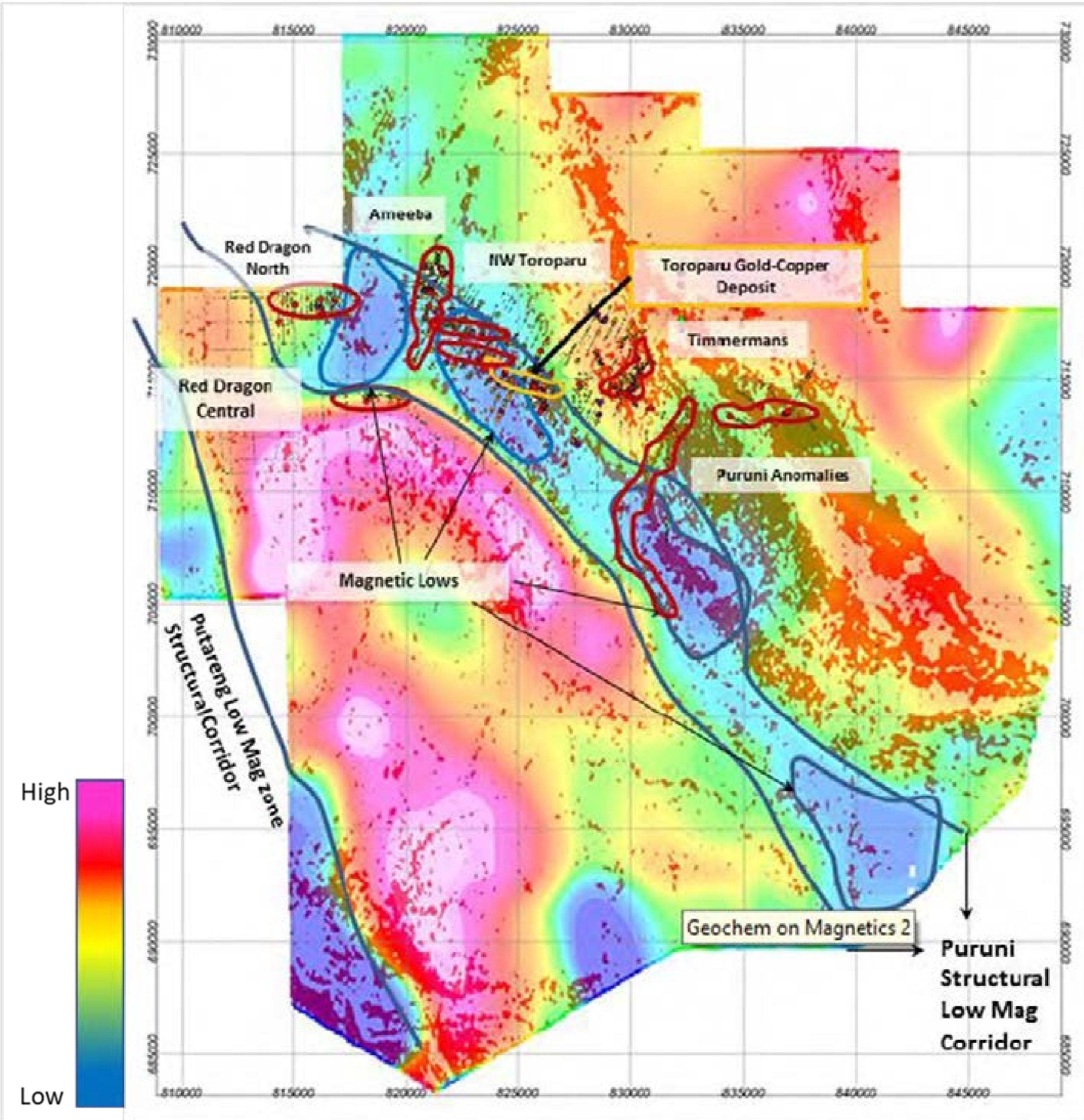
Source: Sandspring's 2011 Annual Exploration Report

Figure 9-2: Location map of regional gold geochemical anomalies. Green = grabbro-norite dykes/sills.

Geophysical work (Source: Sandspring's 2011 Annual Exploration Report

Figure 9-1) in 2010 was subcontracted to Insight Geophysics. The survey consisted of a combined Magnetometer and Gradient Array IP measurements along 200 m spaced lines in an area of 4,800 m x 2,800 m of the Toroparu Deposit (Source: Sandspring's 2011 Annual Exploration Report

Figure 9-3). Results found that chargeability was low over areas of gold-copper mineralization. This discovery led to further targeting and diamond drilling during the 2012 and later 2019-2020 drilling campaigns.



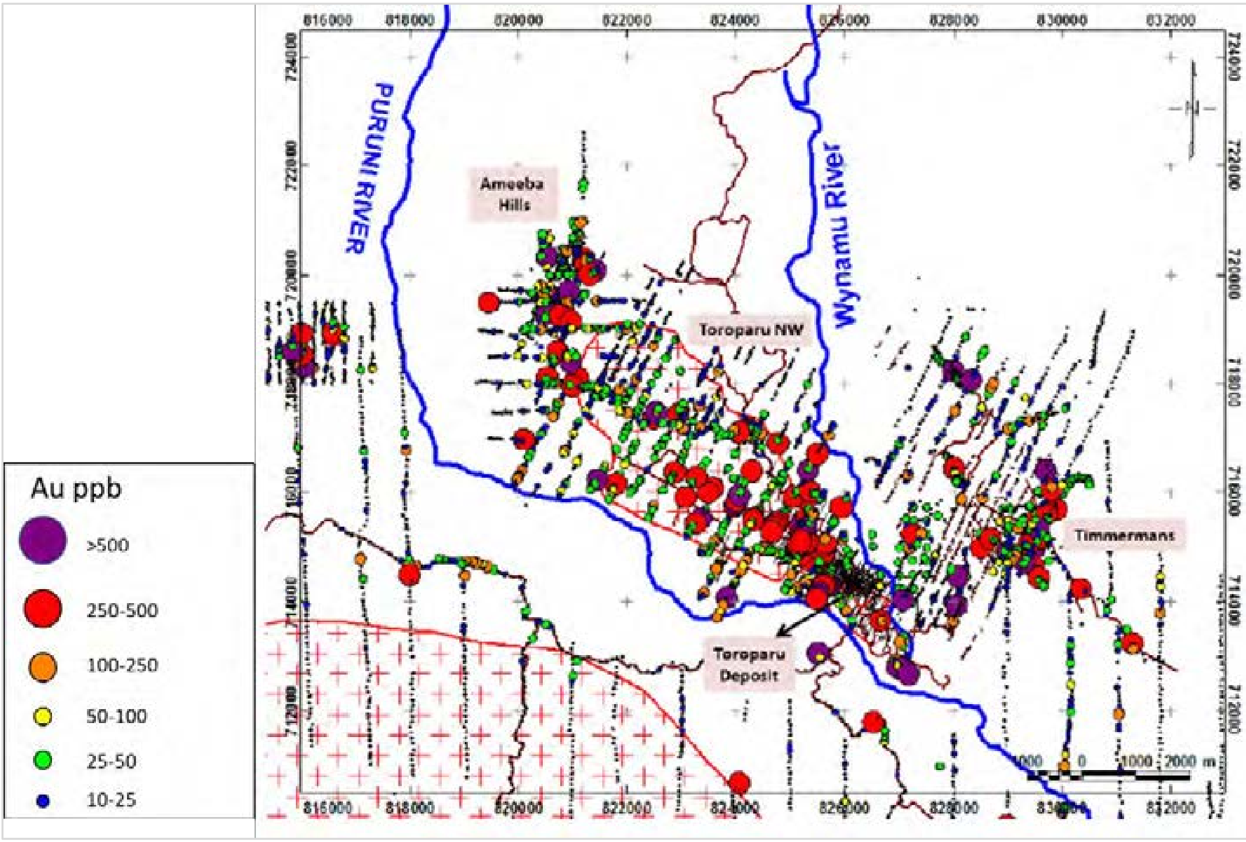
Source: Sandspring's 2011 Annual Exploration Report

Figure 9-3: Gold anomalies overlain on airborne magnetic

9.2.2 2012 Surface Exploration

Geochemical sampling in 2012, added 3,251 samples. Sampling during this program confirmed anomalies detected in the 2011 program (Source: Sandspring's 2011 Annual Exploration Report

Figure 9-4). It also identified three new anomalies, Sona Hill, Sona Hill South, and Majuba located south and southeast of the Toroparu Deposit. It helped to identify geochemical markers for specific lithological units. Middle-Proterozoic mafic sills and dykes seem to be marked by Mercury – Titanium-Vanadium-Cobalt-Silver-Sulphur and Selenium. Red Dragon Granitoids display associations with Thorium-Uranium-Potassium-Aluminum-Titanium-Lead and Barium-Magnesium-Lanthanum-Calcium-Strontium. A significant alteration halo was identified, extending over 20 km to the northwest and southeast from the Toroparu Deposit. This footprint appears to be a result of several significant hydrothermal events.

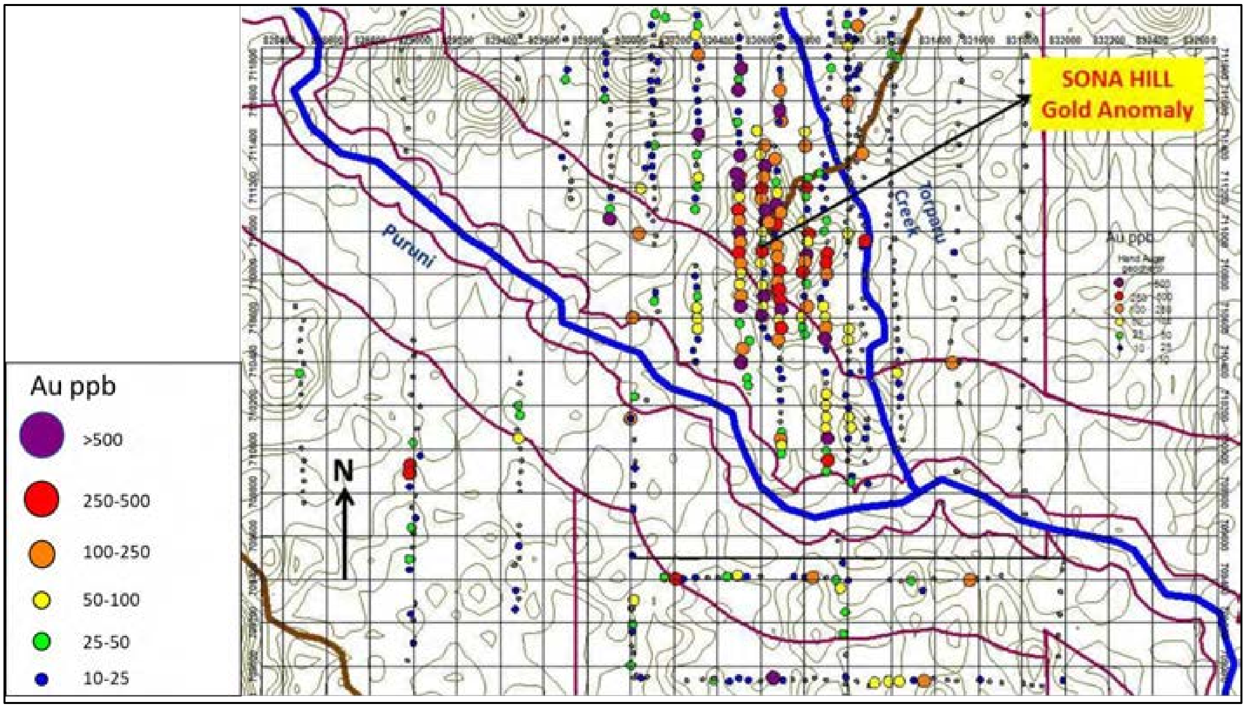


Source: Sandspring's 2011 Annual Exploration Report

Figure 9-4: Geochemical sampling results from 2012-2013 sampling program.

The geophysical ground and airborne magnetic surveys demonstrate that the magnetic high overlaps the mineralized zone (Source: Sandspring's 2012 Annual Exploration Report

Figure 9-5). These survey results suggest an extension of the Sona Hill shear zone to the west and at depth.



Source: Sandspring's 2012 Annual Exploration Report

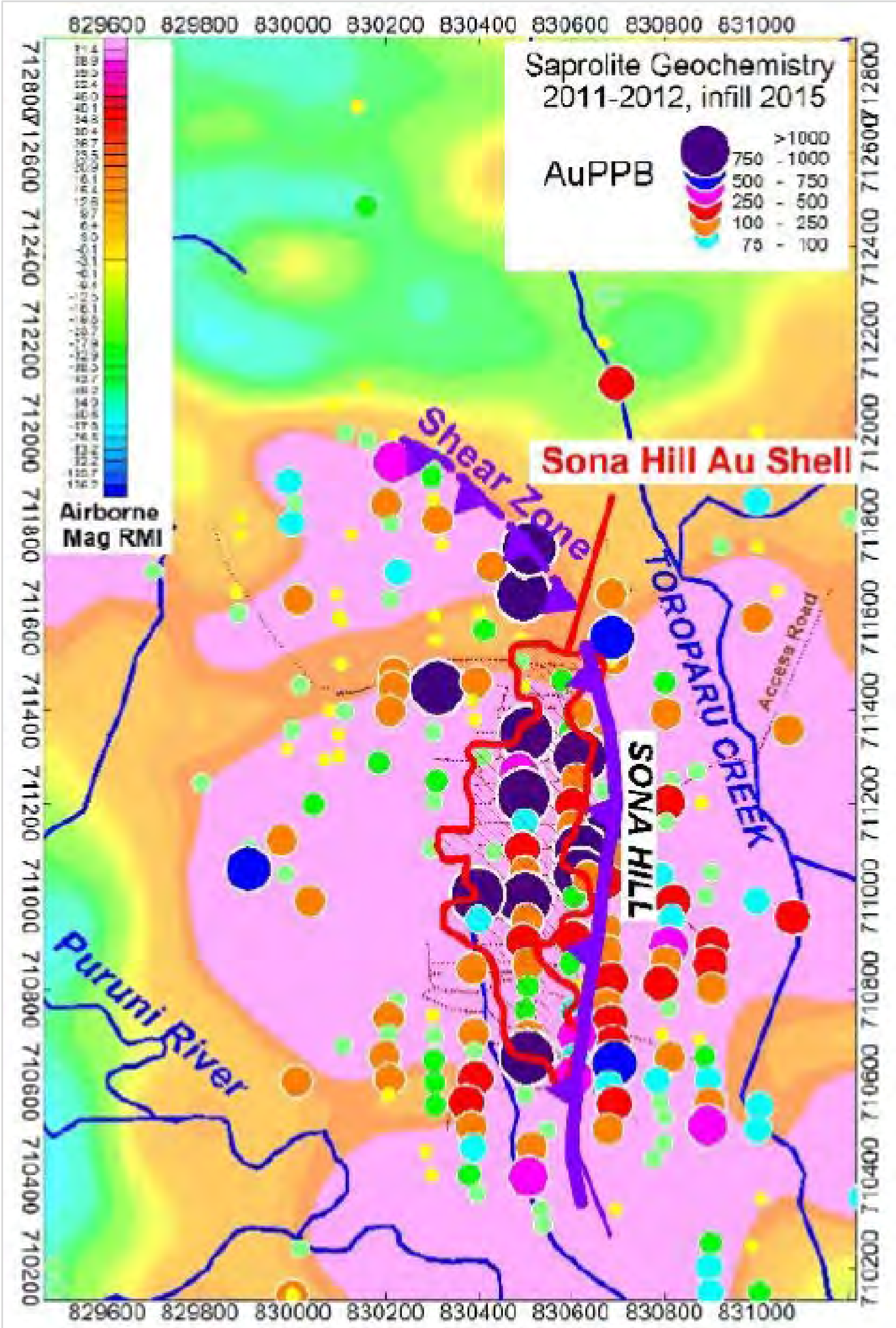
Figure 9-5: Sona Hill Deposit geochemical saprolite gold anomalies

9.2.3 2015-2016 Surface Exploration

The next surface sampling program was not conducted until 2015-2016. Sampling was focused on the Otomung and the Sona Hill Deposit saprolite geochemistry. Previous geophysical surveys over the Otomung area were conducted by TerraQuest Ltd. from December 2010 to February 2011. Airborne geophysics consisted of magnetics, four channel Radiometrics and VLF-Electromagnetics (Source: Sandspring's 2015-2016 Annual Exploration Report

Figure 9-6). Based on these results, a total of 1,038 samples were submitted to Bureau Veritas Commodities Lab for a PR80-250 sample prep and AQ252 30 g assay (37 multi-element analysis) and Aqua Regia digestion and Ultratrace ICP-MS analysis.

Results of the Otomung geochemical sampling were overall weak. The area is mainly flat and marked by numerous swamps causing the saprolite to be covered in clay several metres thick. This could mask the geochemistry of the saprolite in the substratum. A possible elongated intrusive in the centre of the concession was identified. Overall, there is a possibility of a structurally controlled, gold mineralization system along this intrusive feature.



Source: Sandspring's 2015-2016 Annual Exploration Report

Figure 9-6 Airborne magnetic survey with saprolite geochemical sampling overlain

The Sona Hill Deposit geochemical sampling led to the collection of 111 saprolite samples. No significant assays were seen west of the Sona Hill Deposit. New results indicated a possible flexion in the NW of the gold anomaly. Gold and Tellurium demonstrated a good correlation. This surface sampling program led to the possible identification of an extension in mineralization NW of the shear zone's hanging wall. Additional geophysical surveying was completed in 2015. This comprised of an 18 line-km IP and magnetics geophysical survey. The main objective was to map litho-structural differences, particularly the shear zone and identify disseminated sulphide mineralization. Due to the low sulphide content of the Sona Hill Deposit, the chargeability survey did not reveal any significant results. Resistivity did not provide reliable information about the different lithological domains as the intrusives and volcanics display similar mineralogical compositions. Ground magnetics demonstrated three magnetic highs, with the largest clearly overlapping the Sona Hill Deposit mineralization zone. The shape also suggested that the shear zone extended at depth to the west and extended laterally to the northwest and south to south-southeast (Figure 9-7).

Figure 9-7: Airborne magnetic survey conducted over Sona Hill Deposit in 2015-2016

Additional alluvial sampling to acquire information about alluvial mining potential in the river zones adjacent to the Sona Hill Deposit was conducted in 2015. However, barely any alluvial sediments of significance were exposed west of the Sona Hill Deposit. Consequently, alluvial exploration efforts were focused on the Toroparu valley east of the Sona Hill Deposit. A total of 319 samples, including 96 sand/gravel samples, 145 saprolite samples, and 78 concentrates, were submitted to Bureau Veritas Lab in Vancouver. Results were scattered and irregular but showed some elevated grades. If the alluvial surveys continued, it was recommended the investigate the Toroparu valley further to the NW up to the Timmermans prospect. A recommendation of taking larger samples for screen assays and applying wet sieving instead of dry was made.

9.2.4 2017-2018 Surface Exploration

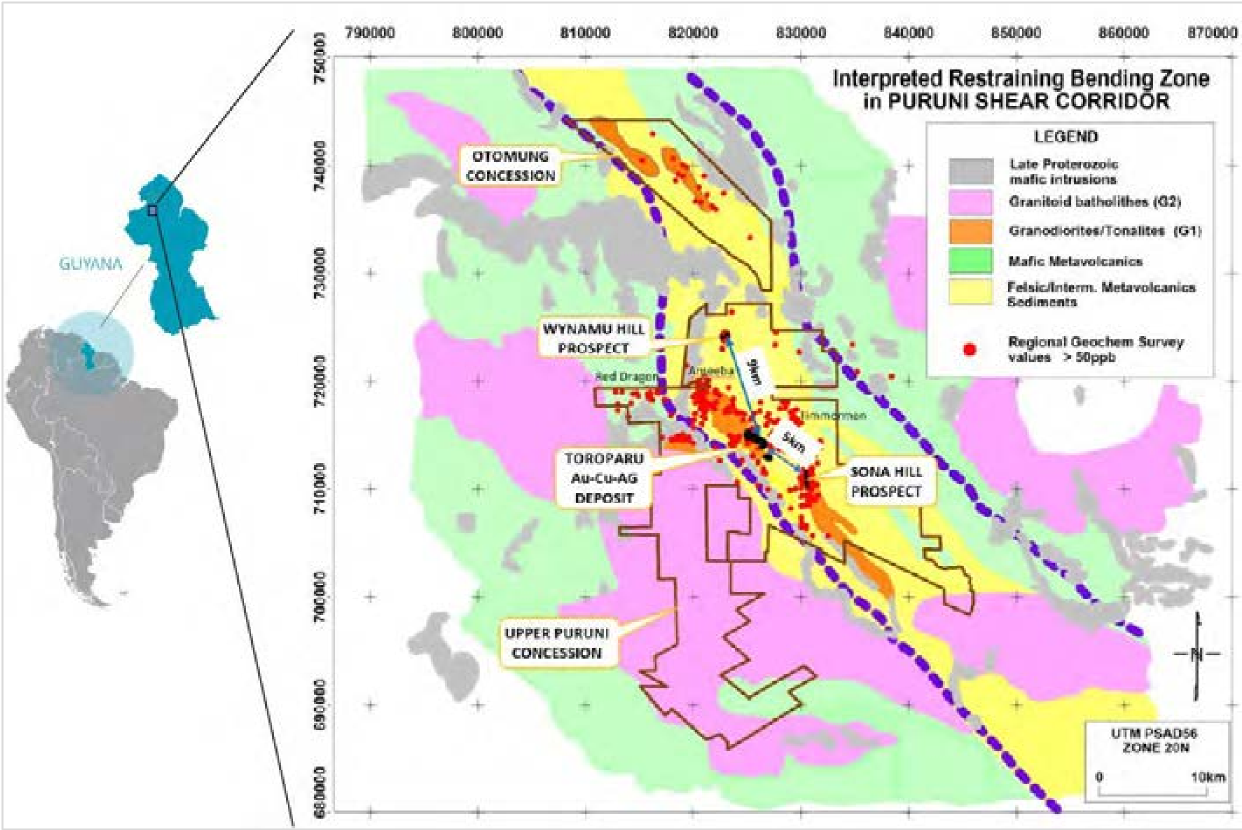
Geochemical sampling in 2018 followed up on the previous 2015-2016 Otomung sampling program. The grid over the previously identified anomalous zones was tightened and 885 saprolite samples were collected. A total of 951 samples (66 QC) were submitted to Bureau Veritas Commodities Lab for a PR80-250 sample prep and AQ252 30 g. Assay, including 37 multi-element analysis, Aqua Regia digestion, and Ultratrace ICP-MS analysis. Gold assay results were weak, not higher than 150 ppb. Results of the exploration work are provided in Source: Sandspring's 2017 – 2018 Annual Exploration Report

Figure 9-8, Source: Sandspring's 2017 – 2018 Annual Exploration Report

Figure 9-9 and Source: Sandspring's 2017 – 2018 Annual Exploration Report

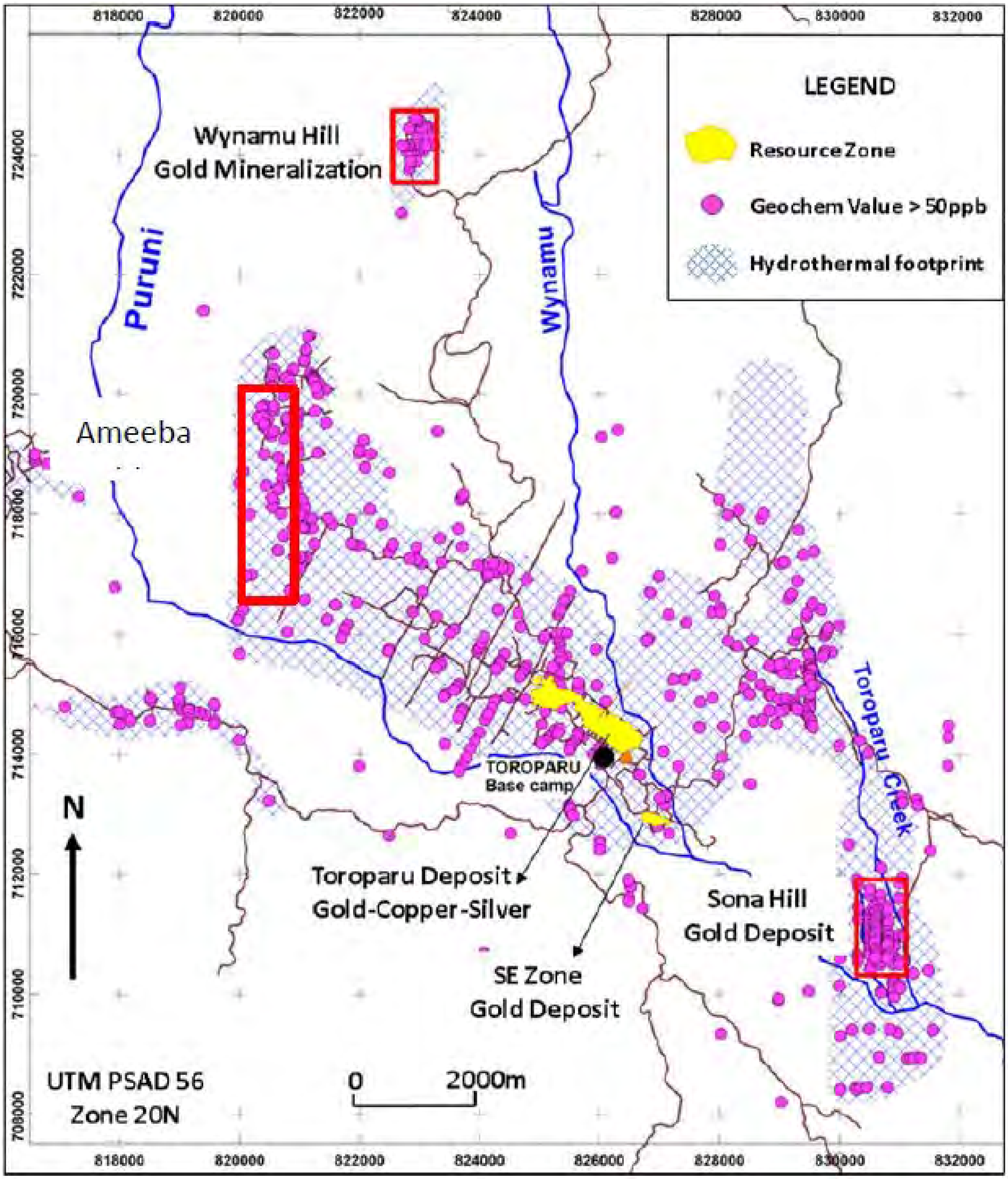
Figure 9-10). Elevated gold values (15 ppb to 50 ppb) displayed a clear 8 km long northwest-southeast oriented trend following the northeastern contact of the interpreted intrusive body identified in 2015/2016 (Source: Sandspring's 2017 – 2018 Annual Exploration Report

Figure 9-8). The gold trend correlates well with anomalous molybdenum features and is marked by some scattered copper values.



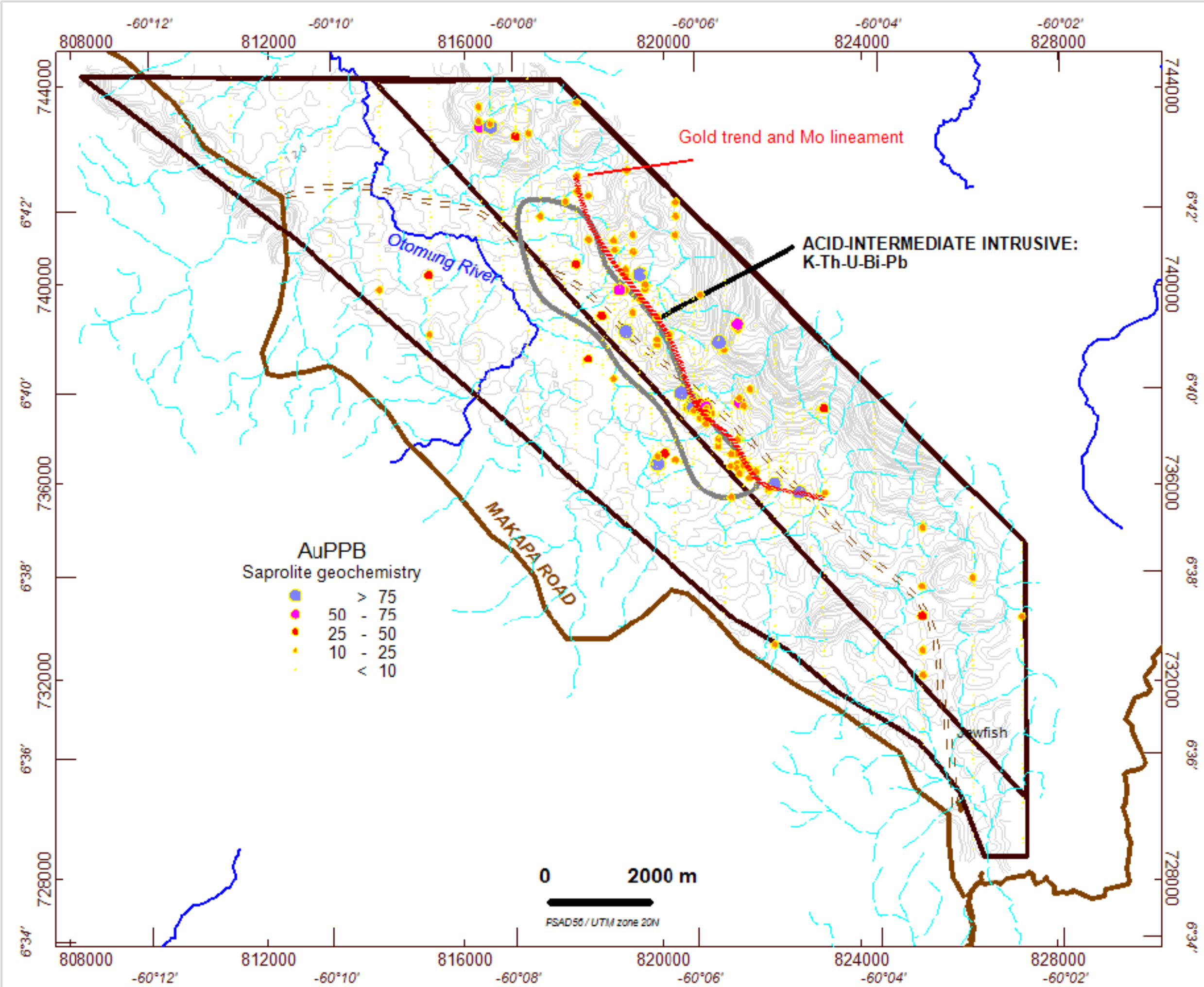
Source: Sandspring's 2017 – 2018 Annual Exploration Report

Figure 9-8 Exploration activities for 2017-2018. Geochemical and geophysical surveying along with planned resource drilling on Sona Hill Deposit. Reconnaissance drilling on Wynamu and geochemical surveying on Otomung



Source: Sandspring's 2017 – 2018 Annual Exploration Report

Figure 9-9: Locations of different prospects on the property marked by gold anomalies as discovered by geochemical saprolite sampling



Source: Sandspring's 2017 – 2018 Annual Exploration Report

Figure 9-10: Otomung concession gold assay results

10 DRILLING

Drilling has occurred at the Project from 2006 through to 2021, directed primarily at the Toroparu Deposit, Main, and SE Areas. At the end of 2021 over 14 years, a total of 215,154 m of resource definition drilling was completed in 788 holes (Figure 10-1).

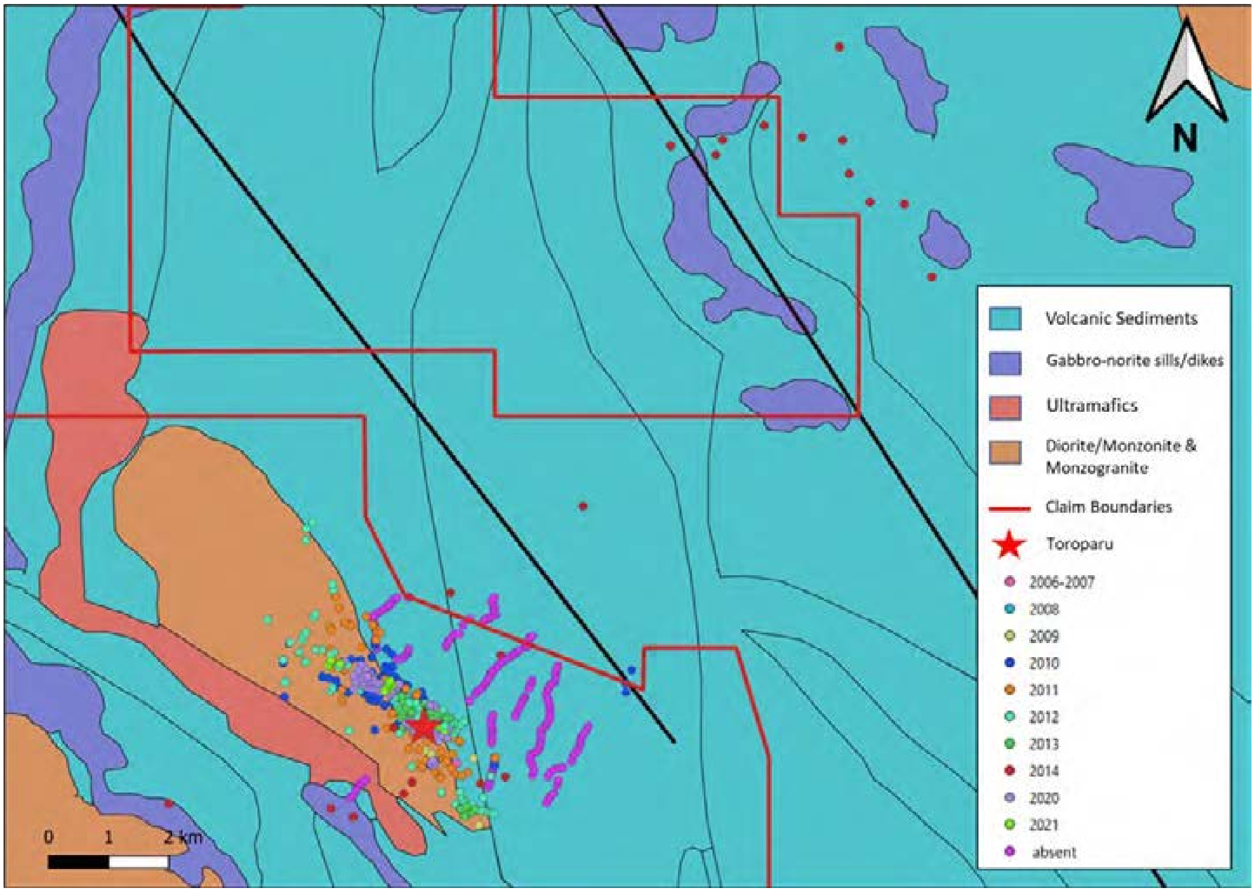


Figure 10-1: Drilling at and surrounding the Toroparu Deposit (Main and SE Areas) by year

Since 2013 most of the exploration drilling throughout the property has been directed toward the Toroparu and Sona Hill deposits and the Wynamu target area. Sona Hill was drilled from late 2015 to early 2018 with 184 diamond drill holes and 21,963 m of drilling: sufficient for resource estimation. Wynamu was drilled with 62 core holes for 6,432.6 m of drilling and further drilling is required to delineate a mineral resource.

In 2020-2021, a drill program consisting of 114 holes and 20,750 m of drilling was conducted on the Toroparu Main and SE areas to test new modelling updates of high-grade structures. Previously the deposit was modelled as a large low-grade, high tonnage system. A new interpretation of the existing data identified cross-cutting high-grade structures throughout the deposit and the results of the new drilling confirmed the new resource model.

Drill hole collar markers for all drill holes on the Project are either steel pipe or steel bars with weld marking of the drill hole ID (Figure 10-2) for hole WMD-56, an exploration hole on the nearby Wynamu exploration target.

Figure 10-2: Typical drill hole collar ID

10.1 2006 – 2018 Diamond Drilling

ETK as the operator in the exploration joint venture with Alphonso initiated drilling on the Toroparu Deposit in December 2006. Several short drilling campaigns were completed during 2007, 2008 and 2009, totalling 20,934 m in 53 holes (Table 10-1). The diamond drilling was carried out by Orbit Garant Drilling Services, Québec, Canada (Orbit).

Table 10-1: Drilling on the Toroparu Deposit, Main Area

Year	Company	Type	Number of Holes	Metreage
Dec 2006-Mar 2007	ETK Inc.	DDH	13	3,480
July-Aug 2007	ETK Inc.	DDH	10	3,748
Apr-May 2008	ETK Inc.	DDH	8	3,555
Aug-Dec 2009	ETK Inc.	DDH	22	10,151
Total			53	20,934

From 2010 to 2018, the Company conducted numerous drilling campaigns with Orbit, including RC and diamond drilling across the Upper Puruni Area (Table 10-2). Initially, with a large focus on the Toroparu Deposit, Main area, and the later focus was on the Sona Hill Deposit in 2018 (Figure 10-3). Other small prospects, such as Wynamu (Source: Sandspring's 2017 – 2018 Annual Exploration Report

Figure 10-4), had some positive drilling results and warrant further exploration in the future.

Table 10-2: Drilling Conducted on the Toroparu, Sona Hill, Wynamu, Otomung, and Ameeba Areas

Year	Location	Type	Number of Holes	Metreage
Jan-Dec 2010	Toroparu Main	DDH	74	42,540
Oct-Dec 2010	Toroparu Main	DDH	5	2,303
Jan-Dec 2011	Toroparu Main	DDH	208	80,455
Jan-Dec 2012	Toroparu Main	DDH	151	34,055
2011	Upper Puruni	RC		15,633
Oct-Nov 2012	Sona Hill	RC	30	2,900
Jul-Dec 2016	Sona Hill	DDH	68	8,084
November 2016	Wynamu	DDH	14	1,127
Nov 2017-Mar 2018	Sona Hill	DDH	58	6,897.5
Total			491	180,287

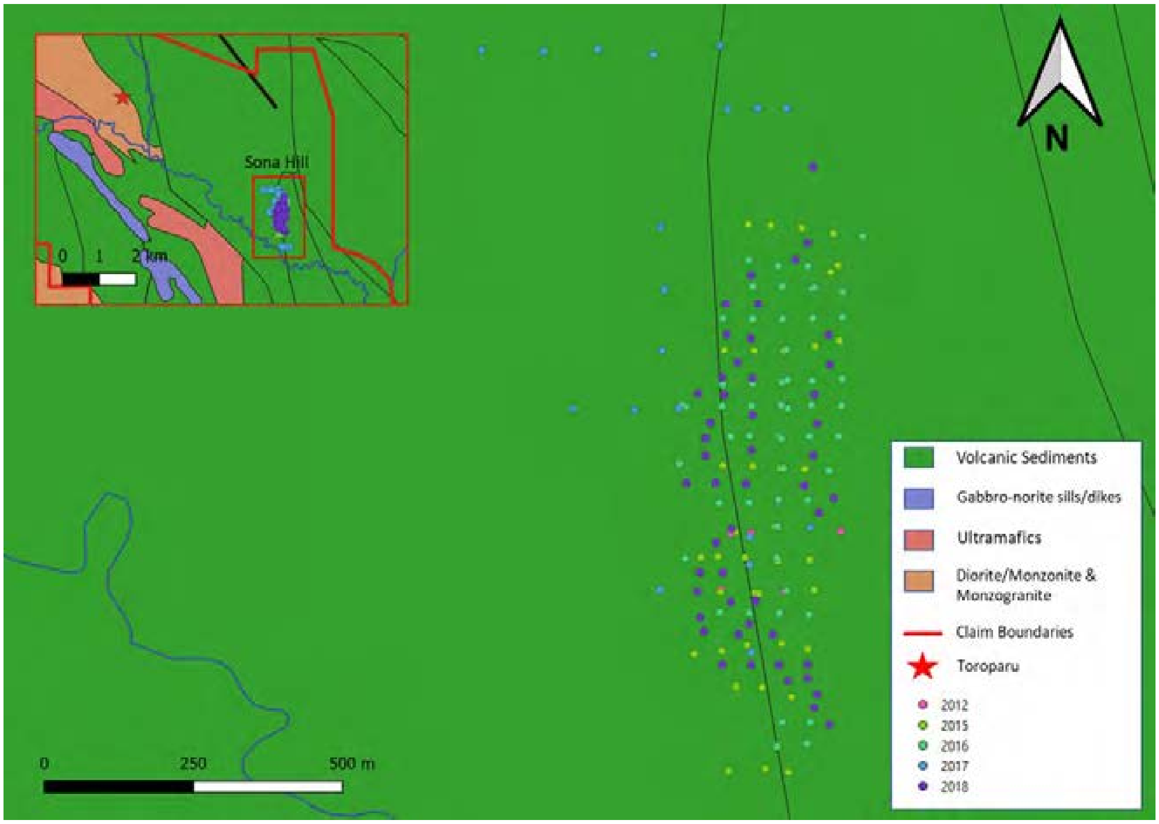
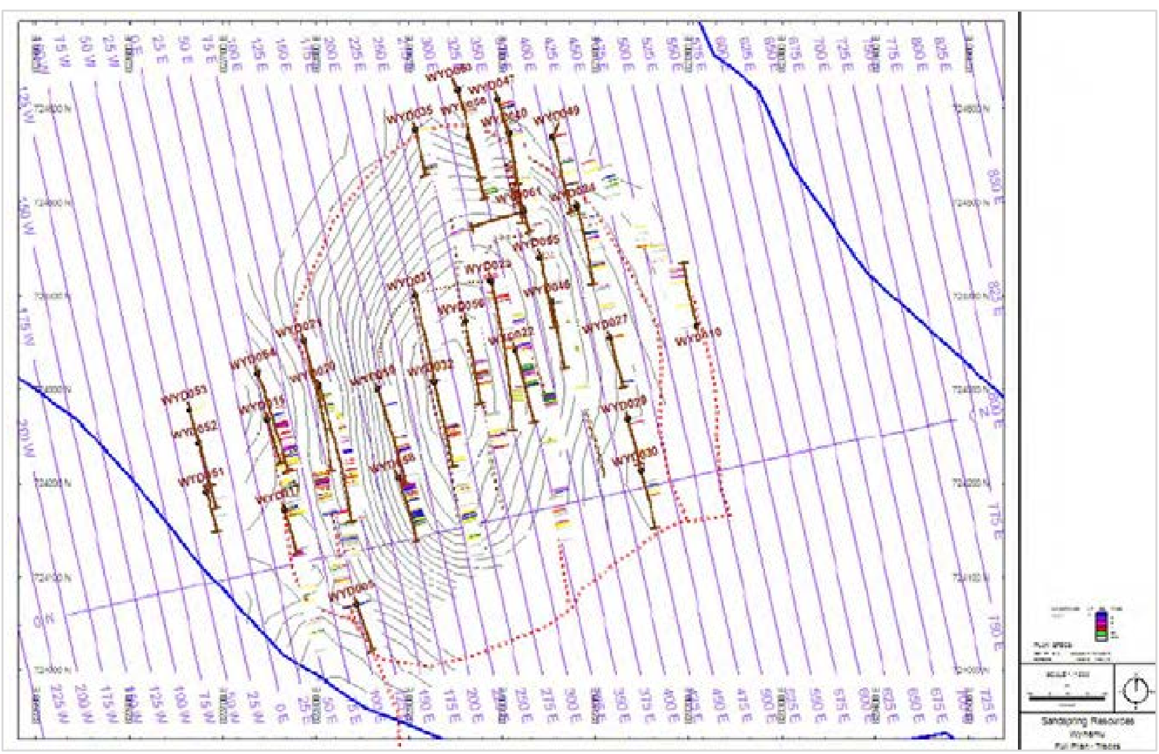


Figure 10-3: Sona Hill drill map with the location of 70 oriented DDH. Drill hole categorized by year drilled



Source: Sandspring's 2017 – 2018 Annual Exploration Report

Figure 10-4: Wynamu hill drill map with oriented drill core

10.2 2020 – 2021 Diamond Drilling

During 2020 and 2021, the Company conducted additional infill drilling focused on the Toroparu Deposit, Main, and NW areas (Table 10-3). Diamond drilling was carried out by Orbit. The focus of this drilling was to confirm the identification of high-grade structures that were re-modelled in 2020. This would allow for the Toroparu Deposit to be mined as both an open pit and underground commodity. Results confirmed the repetitive pattern of the high-grade structures and, consequently, the new model. Drilling to date demonstrated that higher-grade structures have continuity up to 100 m vertically.

Table 10-3: Recent Drilling on the Toroparu Deposit, Main, and SE Areas

Year	Location	Type	Number of Holes	Metreage
Mar-Dec 2020	Toroparu Main	DDH	70	10,146
Jan 2021 – Present	Toroparu Main	DDH	44	10,604
Total		DDH	114	20,750

Table 10-4 and Figure 10-5 show significant intercepts from this drill program outlined in the February 23, 2021, press release. Notably, TPD539 is the furthest step out drill hole, extending the length of the modelled high-grade structures to more than 2.5 km.

Table 10-4: Notable Drill Intercepts from the 2020/2021 Diamond Drilling Campaign (Company Press Release February 23, 2021)

Area	Drill Hole	Length (m)	Grade (g/t)
Toroparu Deposit Main Area	TPD526	4.7	3.1
		9.5	3.4
		4.5	6
	TPD530	7.3	8
		5.3	10.7
	TPD532	3.5	4.4
		3.7	6.3
	TPD533	12	3.31
Toroparu Deposit NW Area	TPD501	13.5	2.66
	TPD509	8	3.6
		3.0	15.63
	TPD539	12.5	4
		3.0	15.63

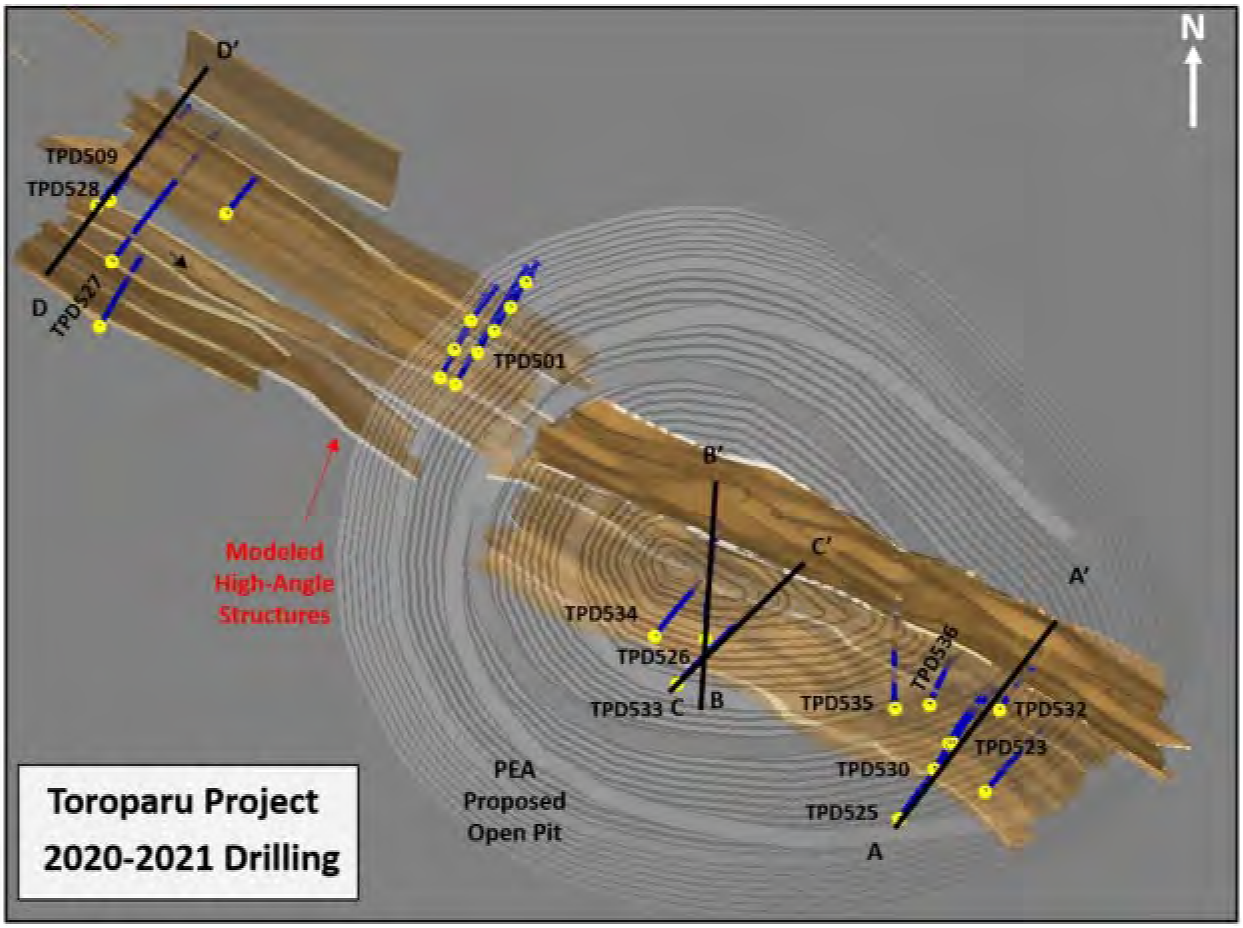


Figure 10-5: Toroparu Deposit high-grade structural model showing the 2020 and 2021 diamond drill holes (Company press release March 22, 2021)

10.3 Drilling Procedures

Drill holes were collared in the field using a mobile handheld GPS, Garmin GPS map 76Cx. Collars were later surveyed using a differential GPS (DGPS) (Trimble GPS Pathfinder ProXRT handheld receiver and NetR9 GNSS base station receiver). All drill hole collar coordinates and elevations were compared and checked to the LIDAR data, which provides a high precision and accuracy topography contour map. Collar coordinates are posted in the digital database to 0.01 m in accuracy for X, Y, and Z UTM coordinates. Core handling procedures consisted of the following:

- Core cleaning; indications of drill depths.
- High resolution photos of whole core.
- Calculating rock quality designation (RQD).
- Indication of sample intervals by the geologist (1.5 m except in zones where lithology justified shorter intervals).
- Systematic bulk density measurements.
- Structural logging consisting of alpha and beta angles on oriented core.
- Detailed geological logging.
- Core cut in half.

- Core sampling.
- Photos of half core in boxes.

The skid-mounted core drill rigs were towed and positioned on the drill collar locations with a small D6 bulldozer. Initial collar surveys of dip and azimuth have been taken using compass (Brunton) measurements for all holes. Downhole surveying was carried out by Orbit, using a digital EZ-Track single/multi-shot instrument at 50 m intervals; the first measurement was done at a standard depth of 15 m to compare with the initial compass measurements at surface. The azimuth provided by the downhole device is referenced to magnetic north; the true azimuth was obtained by making the necessary corrections using the magnetic declination for the area (-15°).

10.4 Core Logging

Geologists checked core boxes at their arrival at the shack and ensured no core was missing. All lithological, structural, and alteration descriptions and the geotechnical and sampling data were digitally coded using a standardized litho-structural and geotechnical code list into the GEMS Logger (Gemcom) software. This data was incorporated into the Project drill hole database. After geological and geotechnical logging, the core is photographed both wet and dry by a digital camera. The digital photos are then stored in a separate core photos database. The diamond drill core is then sampled and cut in half. One half is sent to the lab to be pulverized, and the other half is kept for records in storage.

The main objectives of the core logging and the re-logging exercise were to homogenize geological descriptions and develop a reliable geological model of the Toroparu Deposit, including the definition of geological limits for resource modelling. However, despite improved knowledge of the geological framework, it was difficult to identify clear lithological and/or structural boundaries for the mineralization system. As a result, the resource modellers use gold grade wireframes as shells to constrain mineralization for resource modelling purposes.

A borehole scanning campaign was initiated in the second quarter of 2012: Nine holes (TPD246-247-252 A-255-273-280 A-322-323 A-378) were surveyed by Terratec-Geoservices (Terratec). The following parameters were measured: optical image, ultrasonic image, natural gamma ray emissions, electrical resistivity, IP, hole deviation (azimuth and dip), and total magnetic field. The interpretation from images was done by a Terratec geologist, identifying fractures-foliation contacts, and veins. This survey provided interesting and useful information allowing more detailed structural interpretations.

For the Sona Hill drilling campaign, insitu quartz vein and other structural orientations were derived as measurements on the retrieved scored (oriented) core. Those downhole structural orientations are used in the implicit modelling of geology and grade shells used for Mineral Resource estimation. A total of 45 oriented core holes were completed through the 2016 drilling campaign. Core orientation was performed using the ACT III NQ3 tool, which results in a scored line of the bottom of the core (from hardrock, not saprolite) done at the drill site by the Company's technical team.

Structural measurements are done while logging the core at the campsite and include orientation of fractures, lithology contacts, foliations, and veins/veinlets. The measured alpha and beta values are imported into Geosoft software to convert to true dip angles and dip directions and exported into Excel spreadsheets for inclusion in the database.

10.5 Oriented Core Procedures

In 2015 oriented core was implemented into drill programs to understand structural features and mineralization orientation. Core orientation was performed using the ACT III NQ3 tool. Orientation marking was only possible in the bedrock portion of drilling, not the saprolite. Zones of strong weathering, fracturing, and broken core generally hamper reliable core orientation marking. In most oriented diamond drill holes, core orientation was possible from around 50 m to 60 m of depth and deeper. Core orientation marking was done by the geotechnicians in the field next to the drill. Oriented parts of the core holes were pieced together, and the orientation line was drawn on all core pieces. All structural data are recorded in MS Excel sheets. Subsequently, alpha and beta values were imported in the Geosoft software to be converted in Absolute (true) dip angles and Absolute (True) dip directions.

10.6 Comments on Section 10

In the opinion of the QP, the quantity and quality of the lithological, collar, downhole survey, and specific gravity data collected in the exploration programs are sufficient to support the Mineral Resource Estimate.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Assay Sample Preparation and Analysis

11.1.1 Toroparu Deposit

The sample cutting and bagging are conducted by the Company employees on site. After the core is cut with a diamond saw, half of the core goes into a pre-label sample bag with a tag number and the other half is left in the core box for records (Figure 11-1). For sample preparation, the samples are shipped via airplane to MSA laboratories in Georgetown, Guyana, and from there are shipped to either ACME of Santiago, Chile or ACME of Vancouver, British Columbia, for analysis.

Figure 11-1: Core being halved by diamond saw and sampled with pre-marked sample bags.

The samples are dried, and the entire sample is crushed to better than 80% passing. A 250-gram split is taken and pulverized to better than 85% passing. All samples were analyzed for copper by four-acid digest with atomic absorption spectroscopy (AAS) finish. Most of the samples were analyzed for gold by fire assay method with AAS finish. All samples with results >10 g/t gold were further analyzed using a fire assay method with a gravimetric finish.

11.2 Cyanide Soluble Copper Assays

An update to the Mineral Resource Model for the Project in 2016 included a cyanide soluble copper assessment (CNSolCu) for metallurgical processing and related mine planning purposes. The aim was to determine potential quantities and the spatial distribution of high CNSolCu values that affect the potential processing options and costs.

The evaluation consisted of creating copper and CNSolCu block model grades and their ratios by interpolations using a 973 CNSolCu assay database provided by the Company. Initial investigations comparing CNSolCu with the resource model kriged copper values were not useful; the CNSolCu interpolations with the limited data set cannot be directly compared to the copper block grades interpolated in the Mineral Resource block model. The variation in sampling density and the requirement of extrapolating CNSolCu across the area of interest results in a very dissimilar representation. With the limited sample set, the local CNSolCu estimation is inaccurate, and the modelled grades can only be considered reliable for a global assessment of a metallurgical characteristic. CNSolCu assays were determined from drill core pulps stored at the Project site and analyzed by Bureau Veritas using the LH403 method, which is a cyanide leach and subsequent AAS copper analysis, having a detection limit range of 0.01% to 10% CuCN.

The data was examined for outliers before use in grade interpolation. As the data is sparse, it was important to eliminate the data that did not fit to prevent undue skewing of the subsequent data to be interpolated.

Figure 11-2 shows the plot of copper versus CNSolCu for the entire database provided. There are two populations of data.

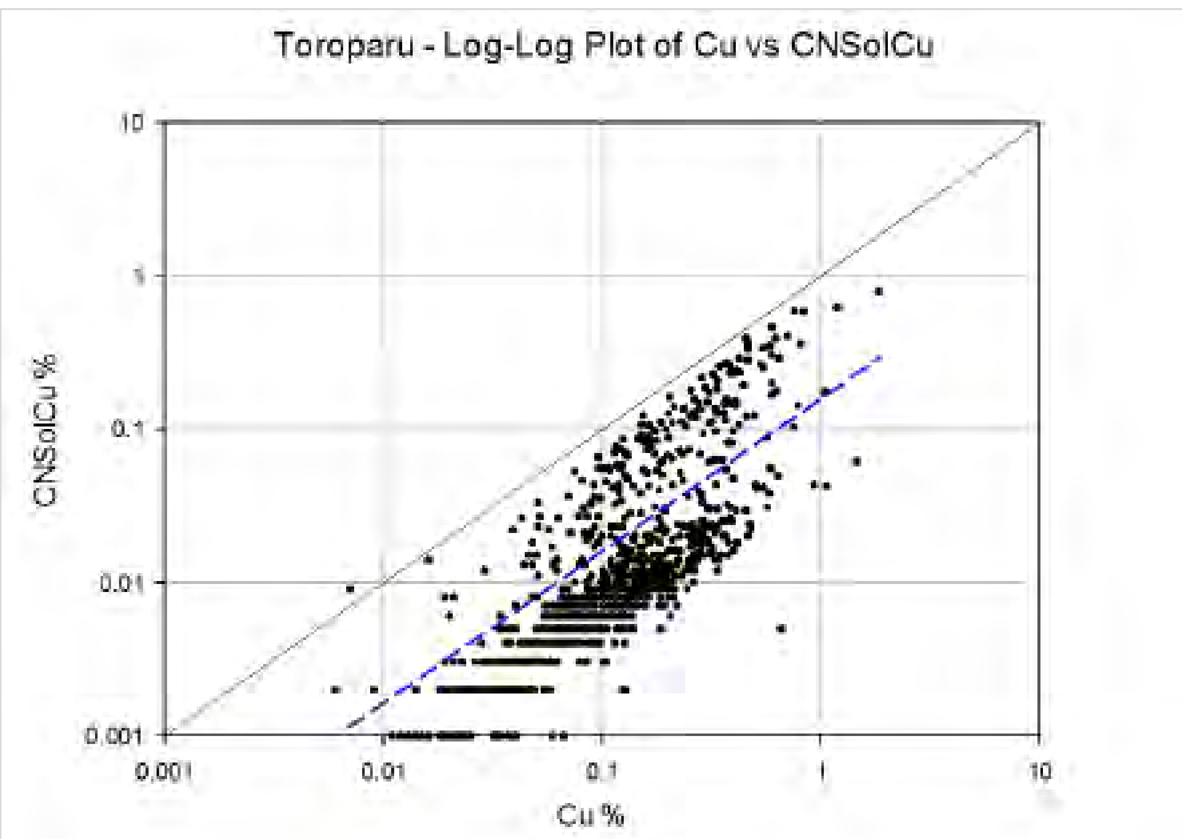


Figure 11-2: Log-Log plot of copper vs. CNSolCu

Figure 11-3 provides a histogram plot of the ratio of CNSolCu to copper. The two populations of data are evident. The geological reasons for the two data populations are not clear.

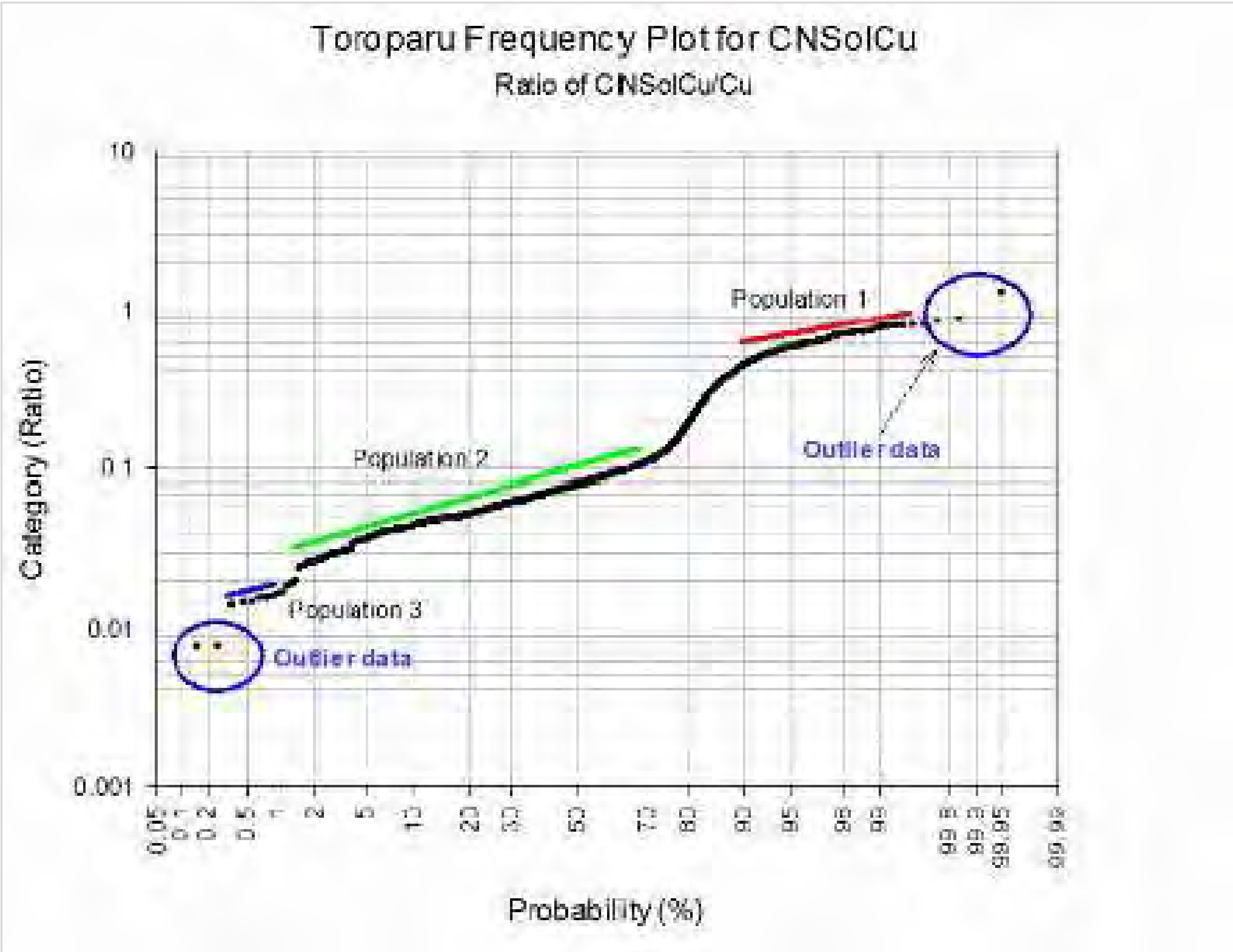


Figure 11-3: Frequency plot of the ratio of CNSolCu

A second frequency plot of copper alone is shown in Figure 11-4, which identifies five high-grade copper assays >1.0% that can be considered outlier data.

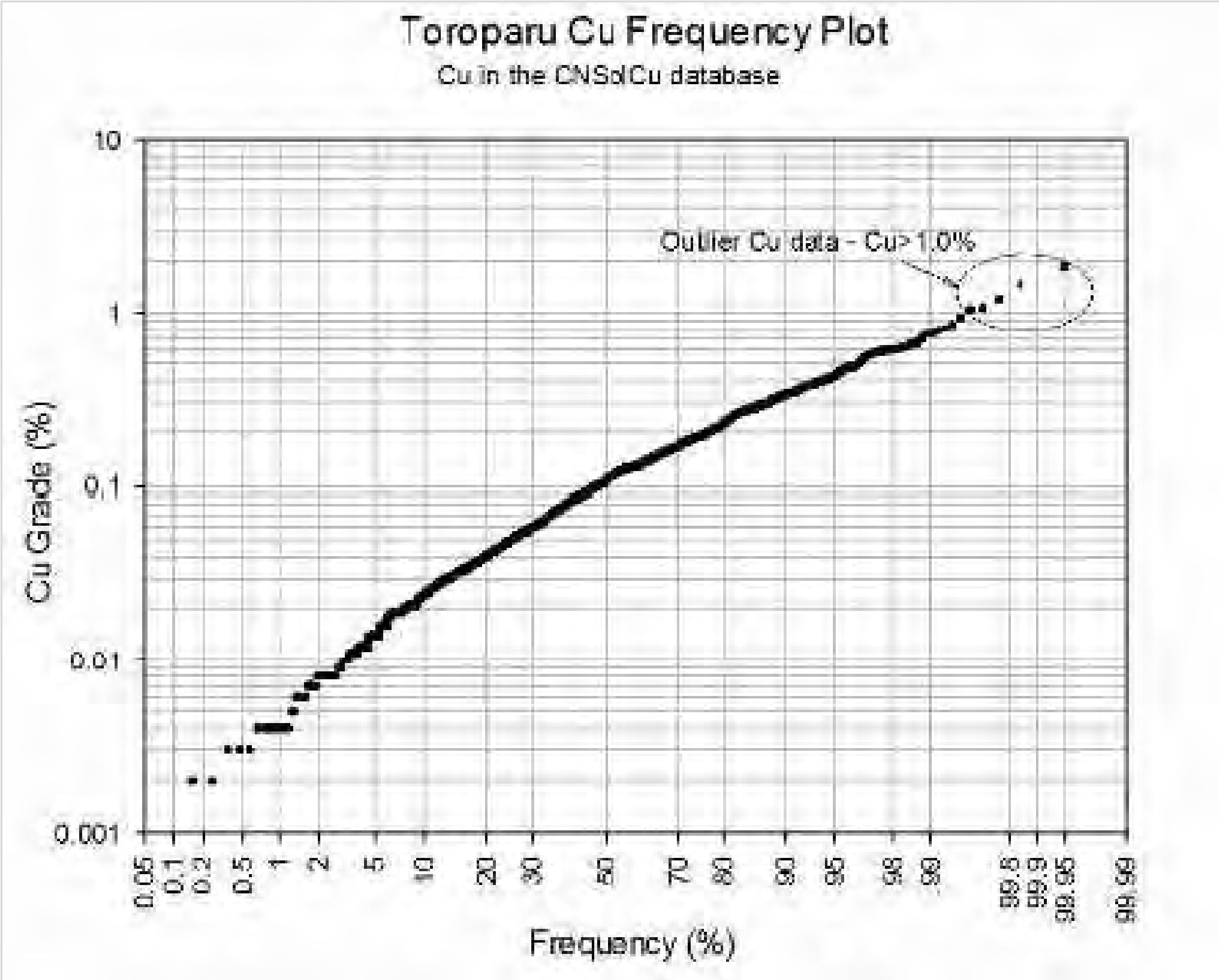


Figure 11-4: Frequency plot of copper

The database was filtered based on the above plots, and the assay outlier data were removed from the database. Ten pairs of data were removed from the database, and the remaining 963 CNSolCu data values were used for grade interpolation. The data removed from the original database before interpolation of CNSolCu values included those with extreme high and low CNSolCu/copper ratios (five pairs) and those with exceptionally high copper values of +1.0% copper (five pair).

It was also noted that for the samples with identified bornite, 18 samples, a total of eight have greater than 50% extractable copper (CNSolCu/copper ratios of +0.5), as would be expected, and most (14) are located at drill depths of 250 m to 450 m. The few bornite-bearing samples with low indicated extractable copper were not considered outlier data, as the samples could indicate local encapsulation of bornite in silica or other minerals. They were not removed from the database.

11.2.1 Conclusions of the 2016 CNSolCu Program

- The model is considered useful for general considerations of a Cyanide Leach only processing for Toroparu Deposit, but with significant limitations.
- The CNSolCu data distribution is suitable for qualitative, comparative purposes at best. It is insufficient for use for quantitative mine planning purposes.

- 600 g/t CNSolCu was identified as the maximum of the problematical range, which would be too difficult to process in a conventional CIL circuit (Smith & Stuart, 2016).

11.3 Specific Gravity Sampling

A total of 6,169 SG measurements exist for the Toroparu Deposit, and 723 SG measurements exist for the Sona Hill Deposit, provided from laboratory measurements. Measurements were taken from DDH samples using the weight in air versus the weight in water method (Archimedes) by applying the following formula:

$$Specific\ Gravity = \frac{Weight\ in\ Air}{(Weight\ in\ Air - Weight\ in\ Water)}$$

11.4 Quality Assurance/Quality Control Programs

QC measures were set to ensure the reliability and trustworthiness of exploration data. These measures include written field procedures and independent verifications of aspects such as drilling, surveying, sampling, assaying, data management, and database integrity. Appropriate documentation of QC measures and regular analysis of QC data is essential as a safeguard for Project data and form the basis for the QA program implemented during exploration.

Analytical QC measures typically involve internal and external laboratory procedures implemented to monitor the precision and accuracy of the sample preparation and assay data. These measures are also important to identify potential sample sequencing errors and to monitor for contamination of samples.

Sampling and analytical QA/QC protocols typically involve taking duplicate samples and inserting QC samples (CRMs and blanks) to monitor the assay results' reliability throughout the drill program. Umpire check assays are typically performed to evaluate the primary lab for bias and involve re-assaying a set proportion of sample rejects and pulps at a secondary umpire laboratory.

11.4.1 Toroparu Deposit, Main Area Pre-2020

Standards

The Company submitted 14 different CRM sample types as part of its QA/QC process, totalling 4,220 CRM before 2020 (Table 11-1). The review of CRM results identified 92 sample swaps or laboratory failures that have been incorrectly identified as members of a different population. Standard CDN-CGS-27 fell primarily within the range of mean ± two standard deviations for gold apart from a few outliers (Figure 11-5), however, copper shows much higher variance than gold (Figure 11-6). CDN-CM-8 mostly fell within the range of mean ± two standard deviations with a few large outliers for gold; again, the copper shows a higher variance than gold (Figure 11-7 and Figure 11-8).

Table 11-1: Toroparu Deposit, Main Area Pre-2020 CRM Result Summary

Standard	Count	Best Value Au (g/t)	Mean Value Au (g/t)	Bias (%)	Best Value Cu (%)	Mean Value Cu (%)	Bias (%)
CDN-CGS-22	91	0.64	0.65	0.01	0.73	0.72	0.01
CDN-CGS-23	75	0.22	0.22	0.00	0.18	0.19	0.01
CDN-CGS-24	408	0.49	0.49	0.00	0.49	0.49	0.00
CDN-CGS-25	531	2.40	2.36	-0.04	2.19	2.18	-0.01
CDN-CGS-27	540	0.43	0.45	0.01	0.38	0.39	0.01
CDN-CGS-29	268	0.23	0.23	0.00	0.58	0.58	0.00
CDN-CM-04	67	1.18	1.14	-0.04	0.51	0.50	0.01
CDN-CM-08	474	0.91	0.90	-0.01	0.36	0.37	0.01
CDN-CM-13	343	0.74	0.75	0.01	0.79	0.80	0.01
CDN-CM-14	346	0.79	0.80	0.01	1.06	1.06	0.00
CDN-CM-15	566	1.25	1.27	0.02	1.28	1.26	-0.02
CDN-CM-19	44	2.11	2.20	0.09	2.02	2.14	0.12
CDN-CM-21	28	0.47	0.48	0.01	0.53	0.54	0.01
CDN-ME-07	439	0.22	0.21	-0.01	0.23	0.23	0.00

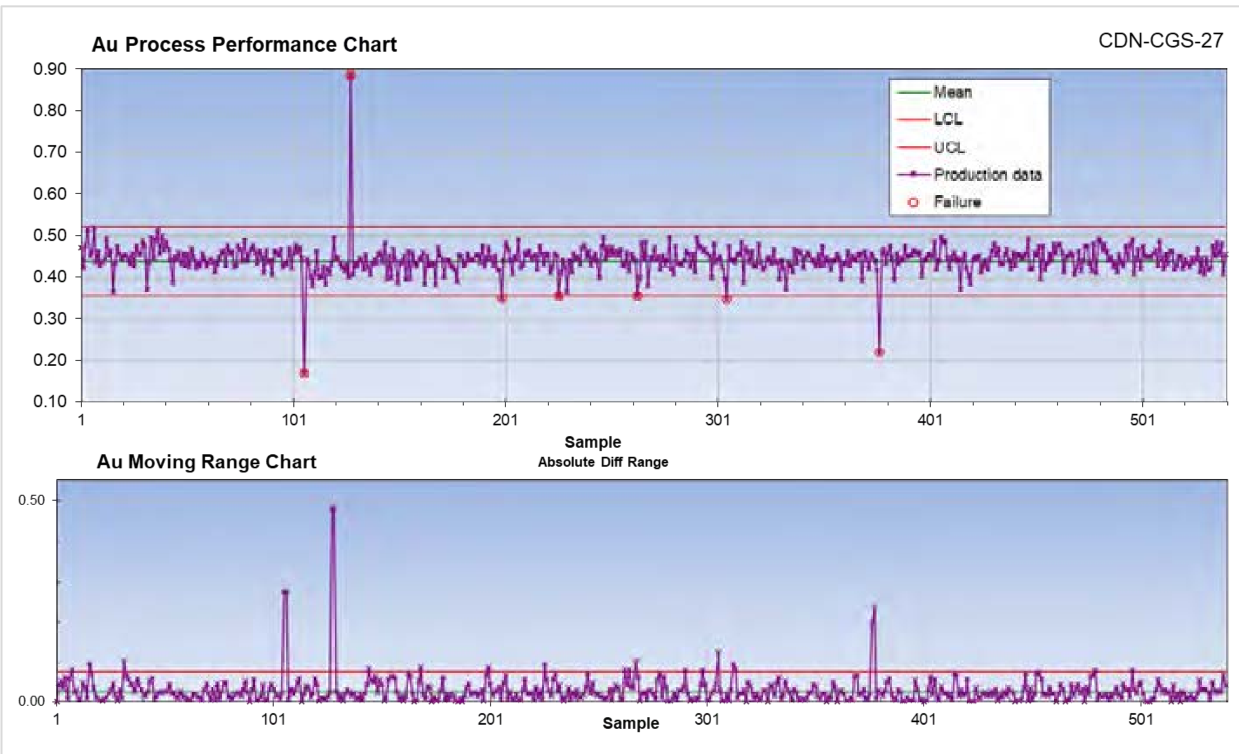


Figure 11-5: Toroparu Deposit Standard CDN-CGS-27 gold (g/t)

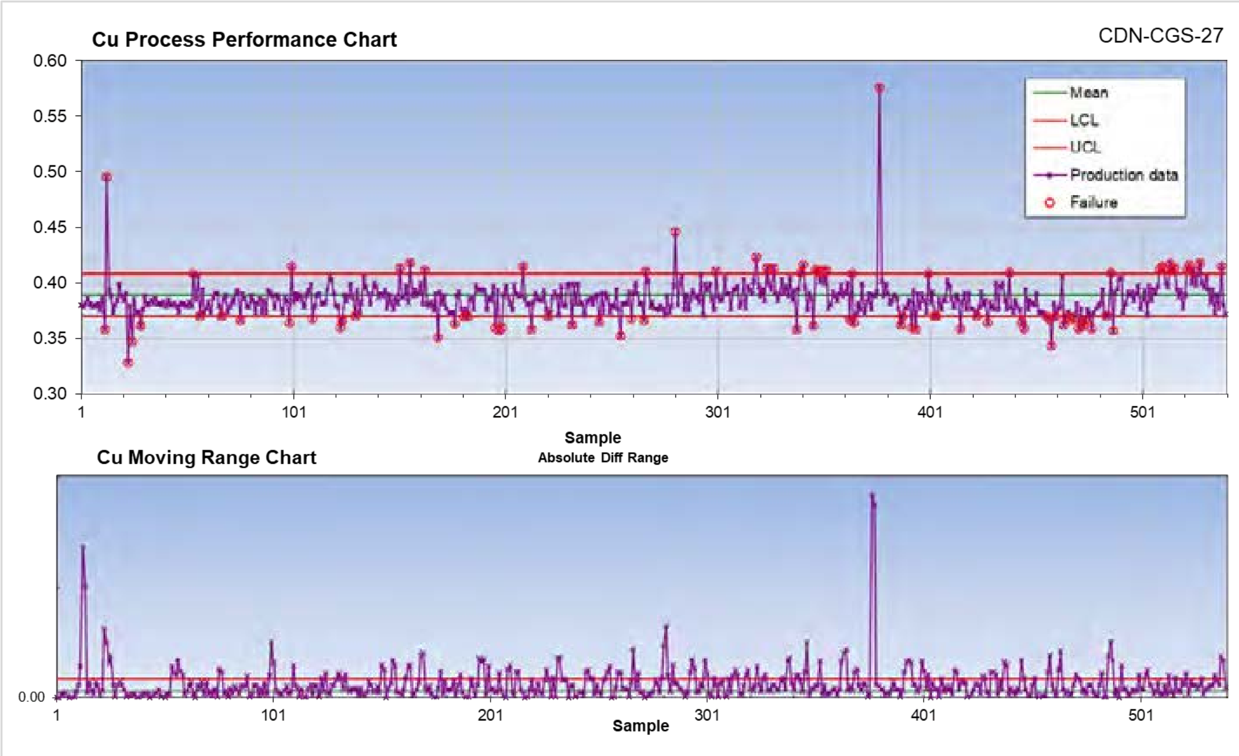


Figure 11-6: Toroparu Deposit Standard CDN-CGS-27 copper (%)

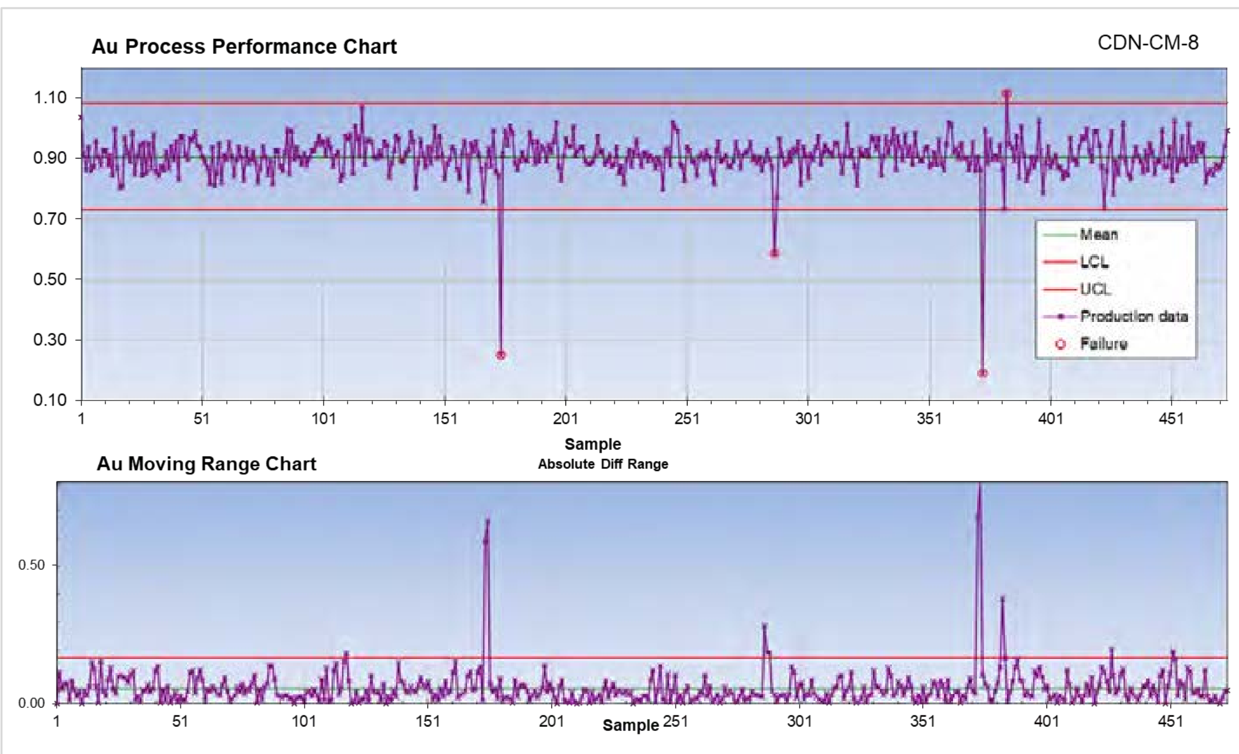


Figure 11-7: Toroparu Deposit Standard CDN-CM-8 gold (g/t)

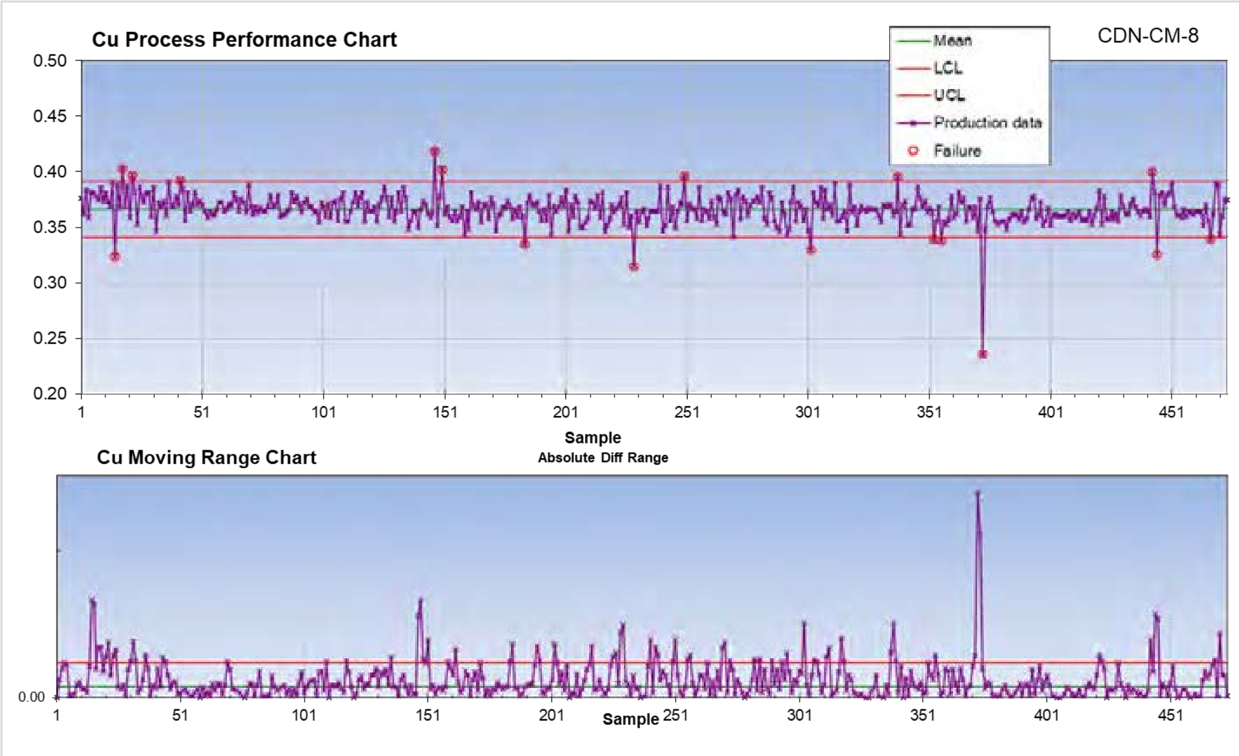


Figure 11-8: Toroparu Deposit Standard CDN-CM-8 copper (%)

<u>Blanks</u>

The Company submitted 2,784 coarse blanks before 2020 as part of its QA/QC process. One coarse blank was used and tested for gold and copper (Figure 11-9 and Figure 11-10). The blank for gold shows a large variance within a cluster area and may indicate short-term calibration issues at the lab. The blank for copper also shows high variance; however, these are more spread out. No significant contamination is evident, as there is no obvious correlation between the blank values and those samples immediately preceding.

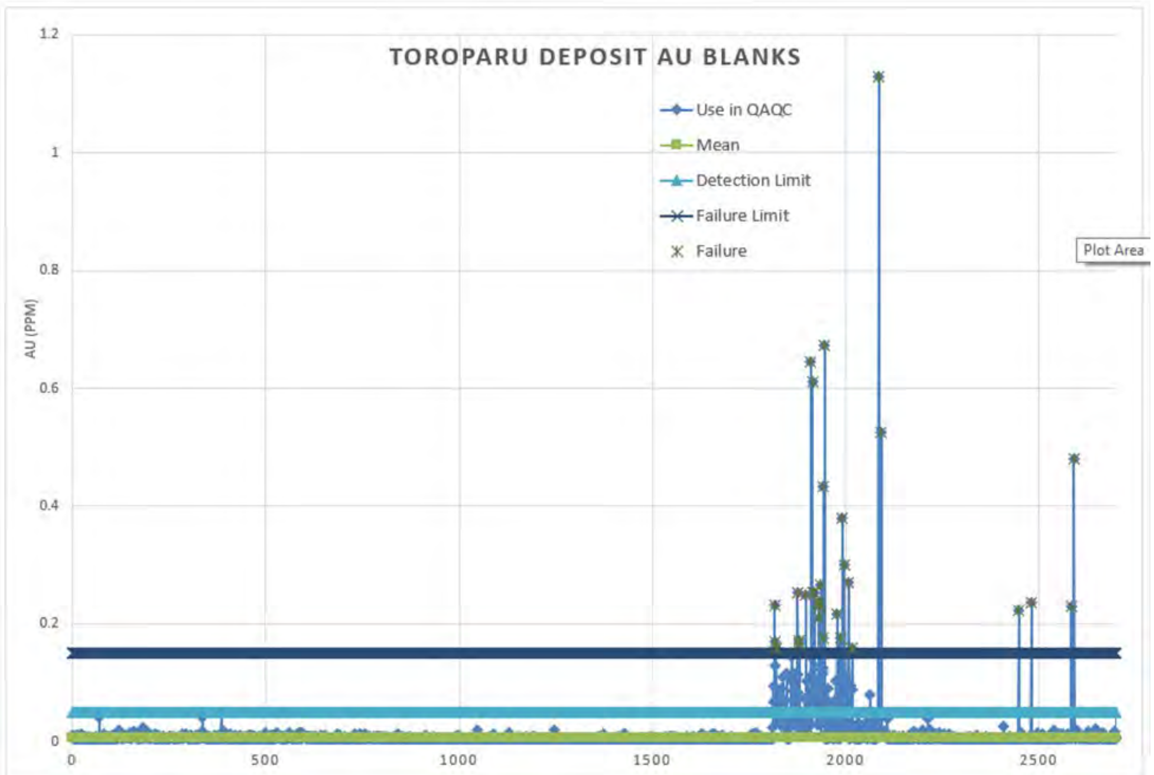


Figure 11-9: Toroparu Deposit coarse blanks for gold (g/t)

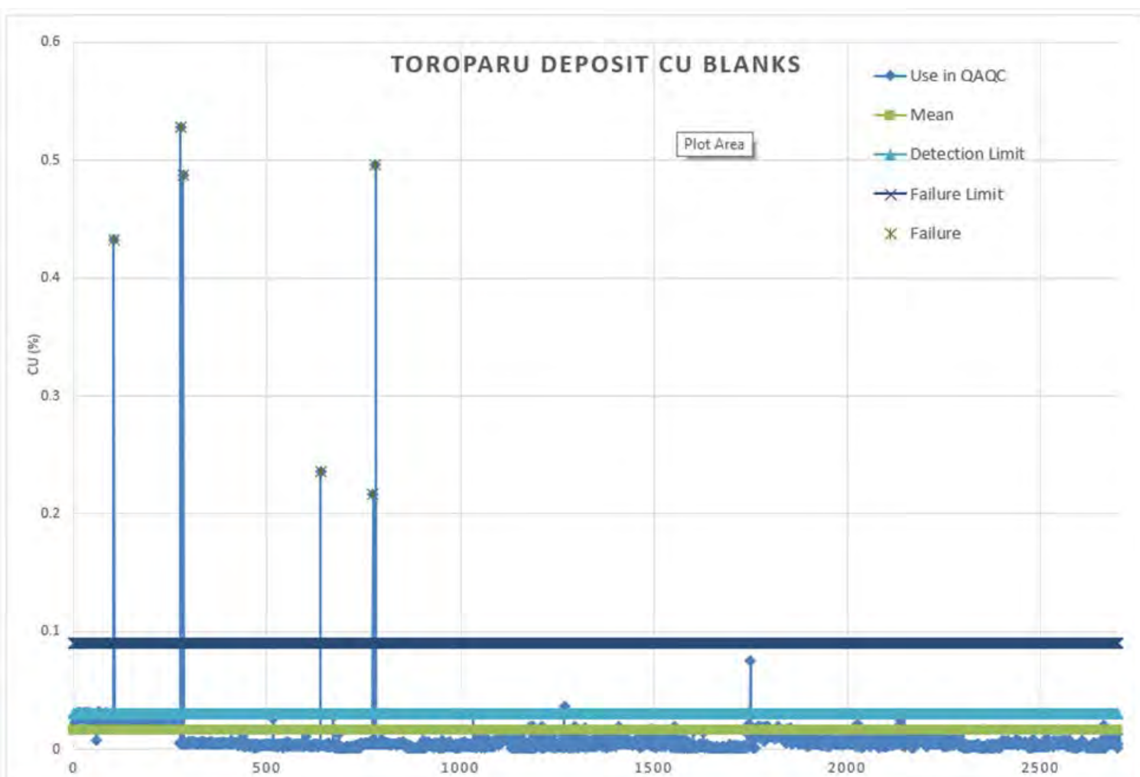


Figure 11-10: Toroparu Deposit coarse blanks for copper (%)

Field and Laboratory Duplicates

The Company submitted 1,252 field duplicates, and the lab submitted 3,135 pulp duplicates as part of their QA/QC process from before 2020. The field duplicates show some variance for gold and copper (Figure 11-11 and Figure 11-12). The lab pulp duplicates show good agreement for gold and copper (Figure 11-13 and Figure 11-14).

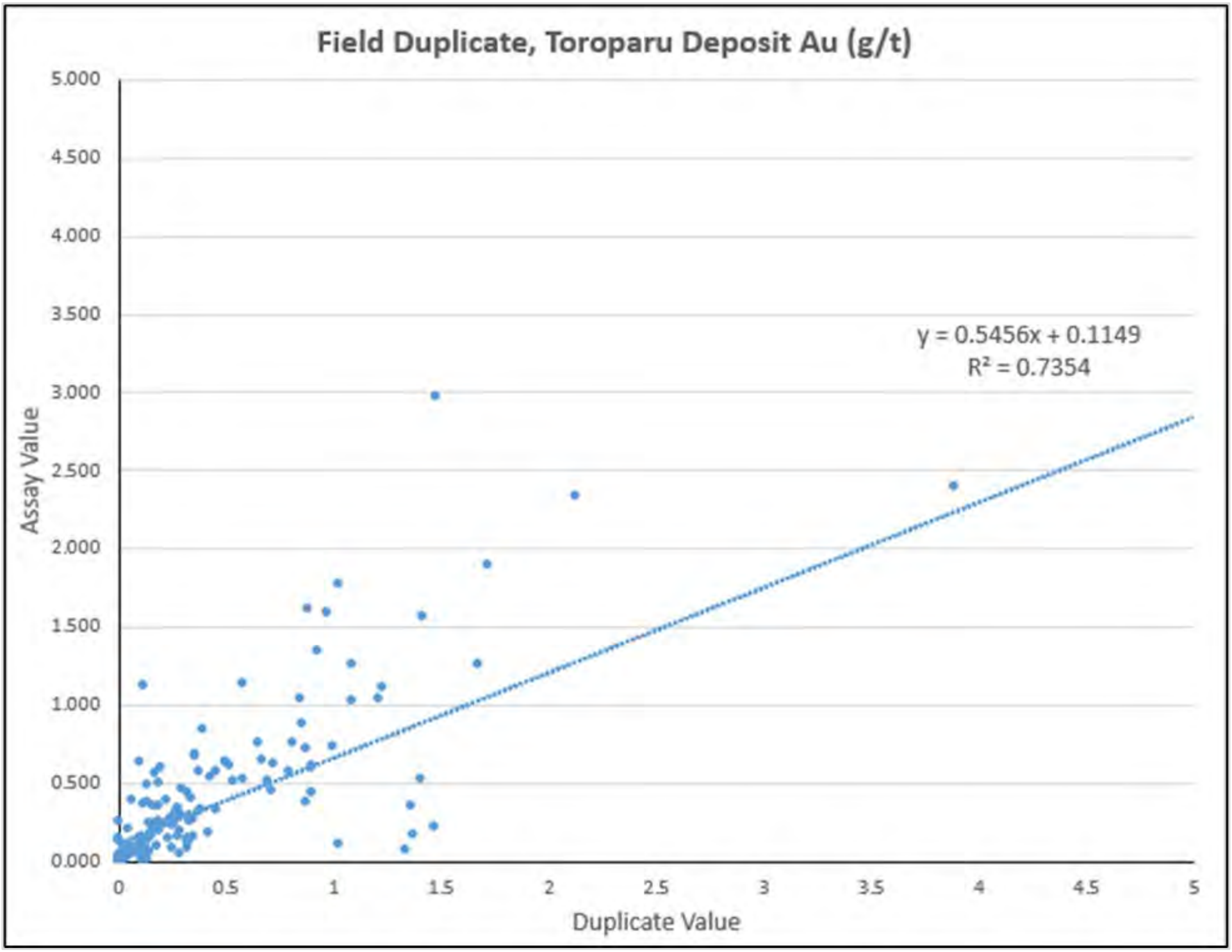


Figure 11-11: Toroparu Deposit field duplicate for gold (g/t)

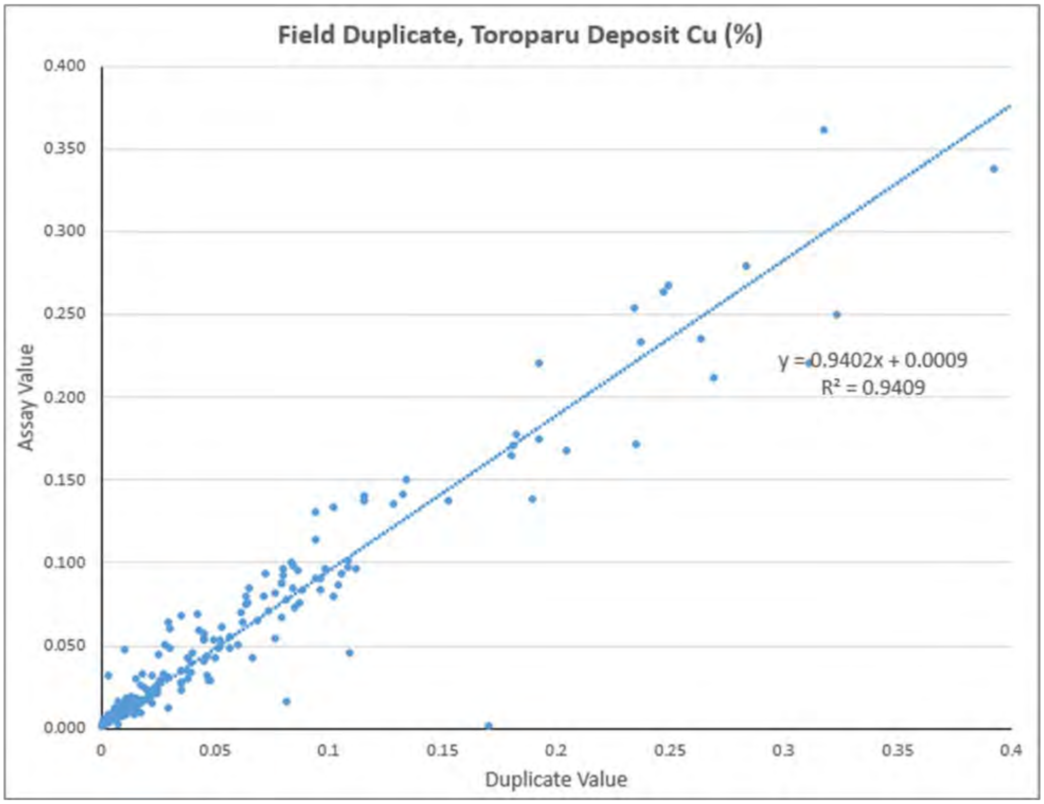


Figure 11-12: Toroparu Deposit field duplicates for copper (%)

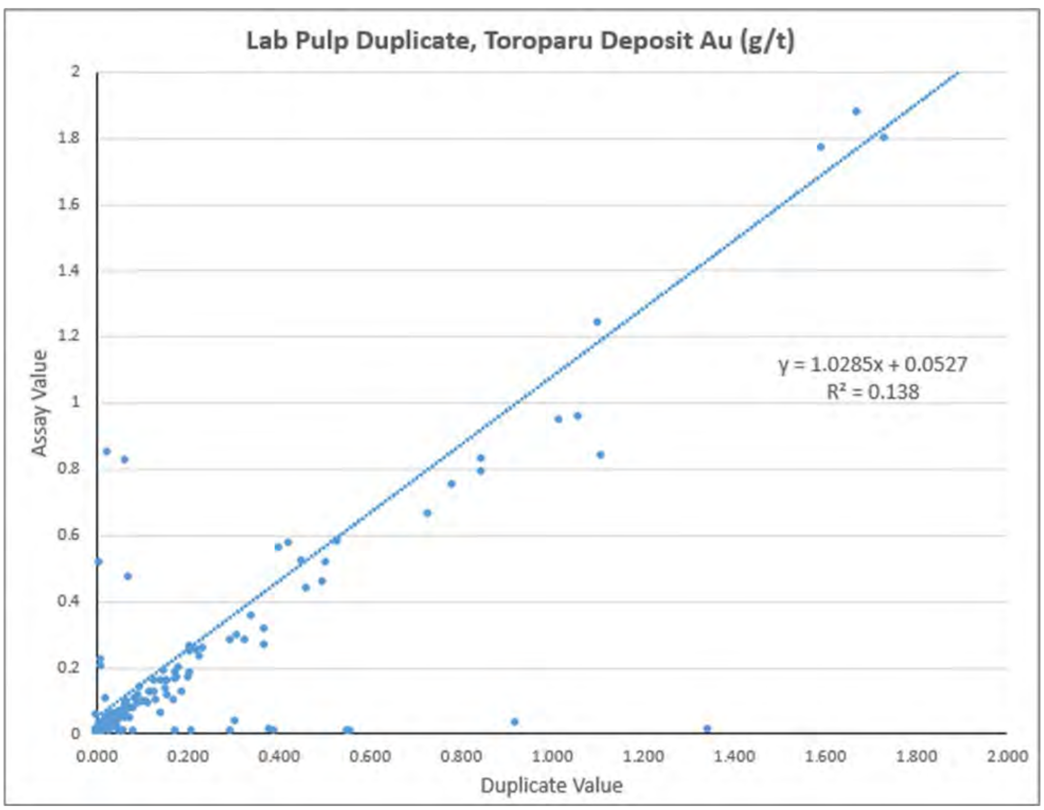


Figure 11-13: Toroparu Deposit lab pulp duplicates for gold (g/t)

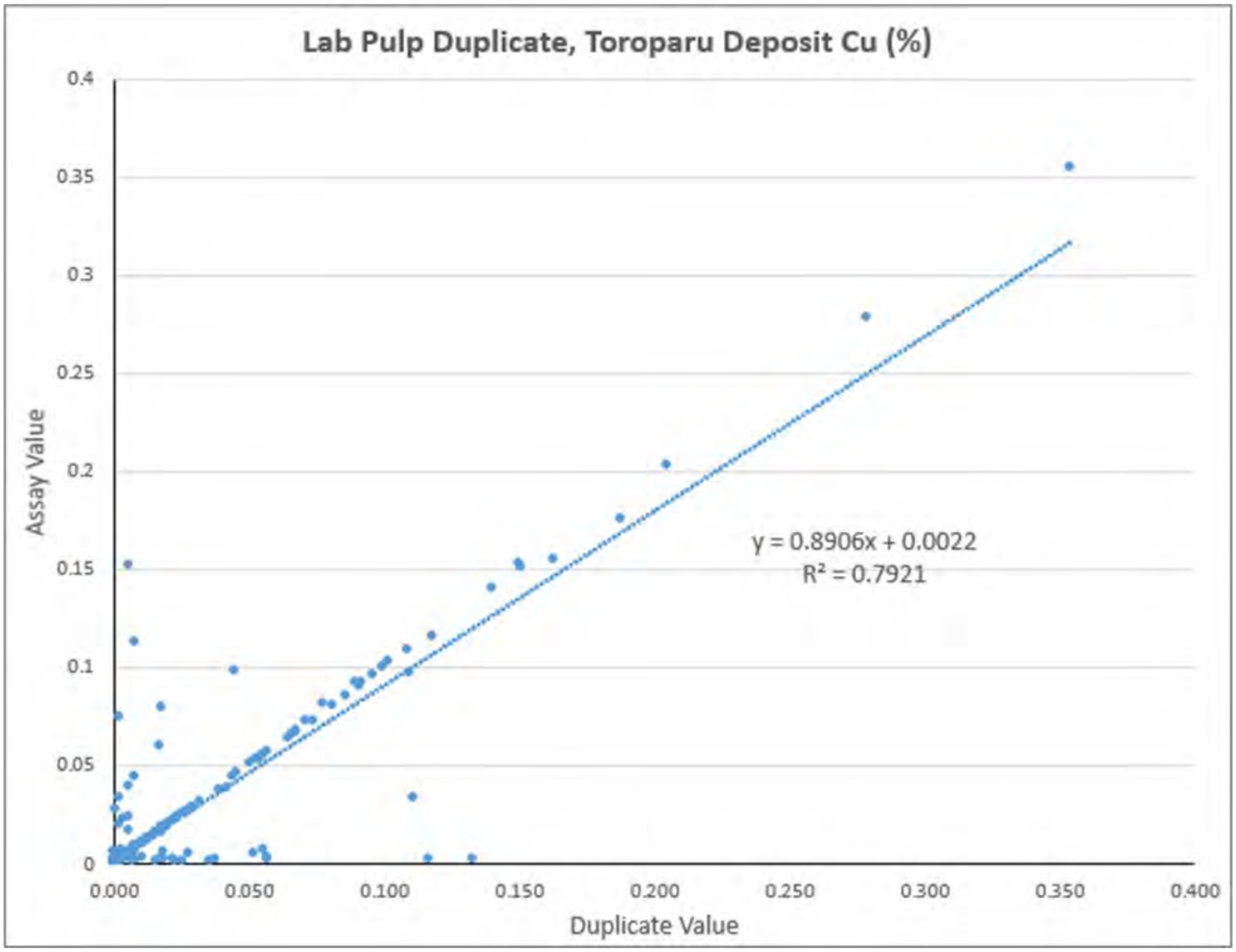


Figure 11-14: Toroparu Deposit lab pulp duplicates for copper (%)

11.4.2 Toroparu Deposit, Main Area 2020 to 2021

Standards

The Company submitted five different CRMs as part of its QA/QC process with a total of 622 CRM from 2020 to 2021 for silver, gold, and copper (Table 11-2). The standard CDN-CM-37 shows good agreement and fell mostly within the range of mean ± two standard deviations for silver, gold, and copper (Figure 11-15, Figure 11-16 and Figure 11-17). The standard CDN-CM-40 fell mostly within the range of mean ± two standard deviations for silver, gold, and copper, except silver and gold show few major outliers (Figure 11-18, Figure 11-19 and Figure 11-20).

Table 11-2: Toroparu Deposit, Main Area 2020-2021 CRM Result Summary

Standard	Count	Best Value Ag (g/t)	Mean Value Ag (g/t)	Bias (%)	Best Value Au (g/t)	Mean Value Au (g/t)	Bias (%)	Best Value Cu (%)	Mean Value Cu (%)	Bias (%)
CDN-CM-15	57				1.253	1.328	0.075	1.28	1.269	0.011
CDN-CM-37	78	1.17	1.18	0.01	0.171	0.178	0.007	0.212	0.214	-0.002
CDN-CM-38	125	6	6.03	0.03	0.942	0.945	0.003	0.686	0.678	0.008
CDN-CM-39	180	5.3	5.18	-0.12	0.687	0.703	0.016	0.538	0.529	0.009
CDN-CM-40	182	18	17.441	-0.559	1.31	1.326	0.016	0.561	0.57	-0.009

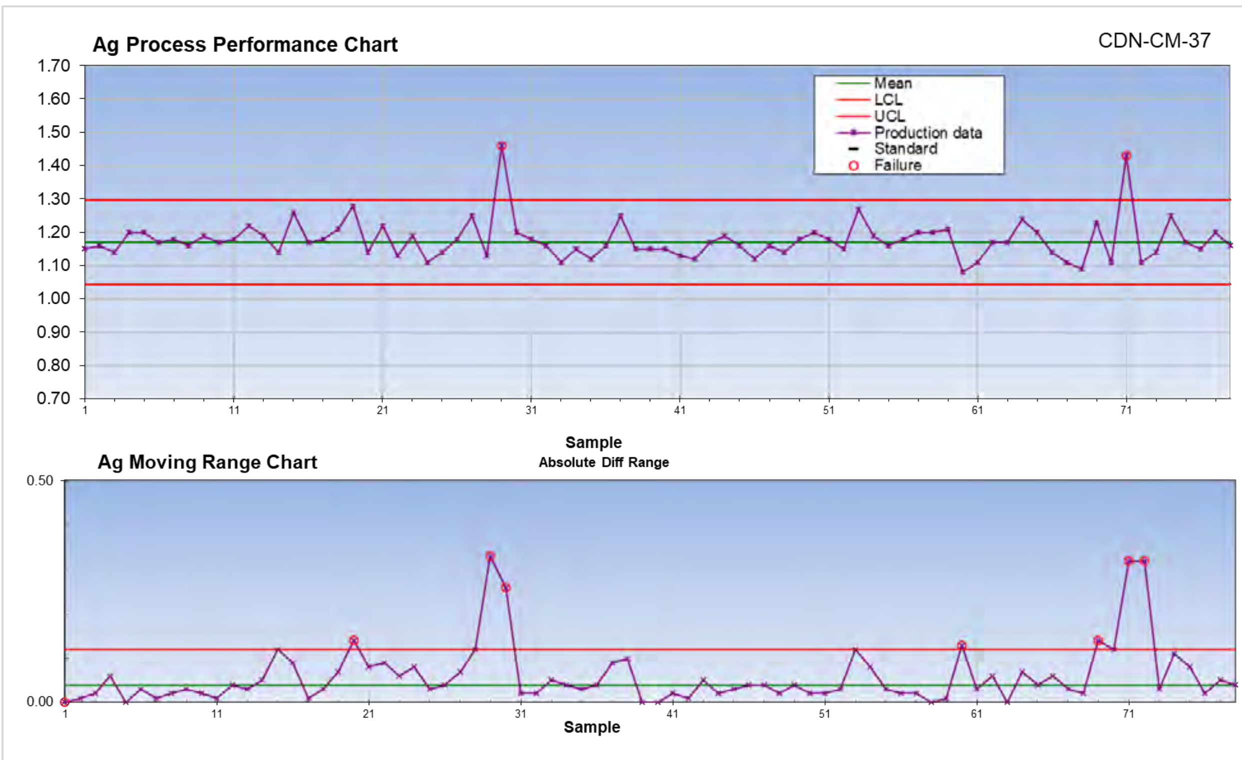


Figure 11-15: Toroparu Deposit Standard CDN-CM-37 silver (g/t)

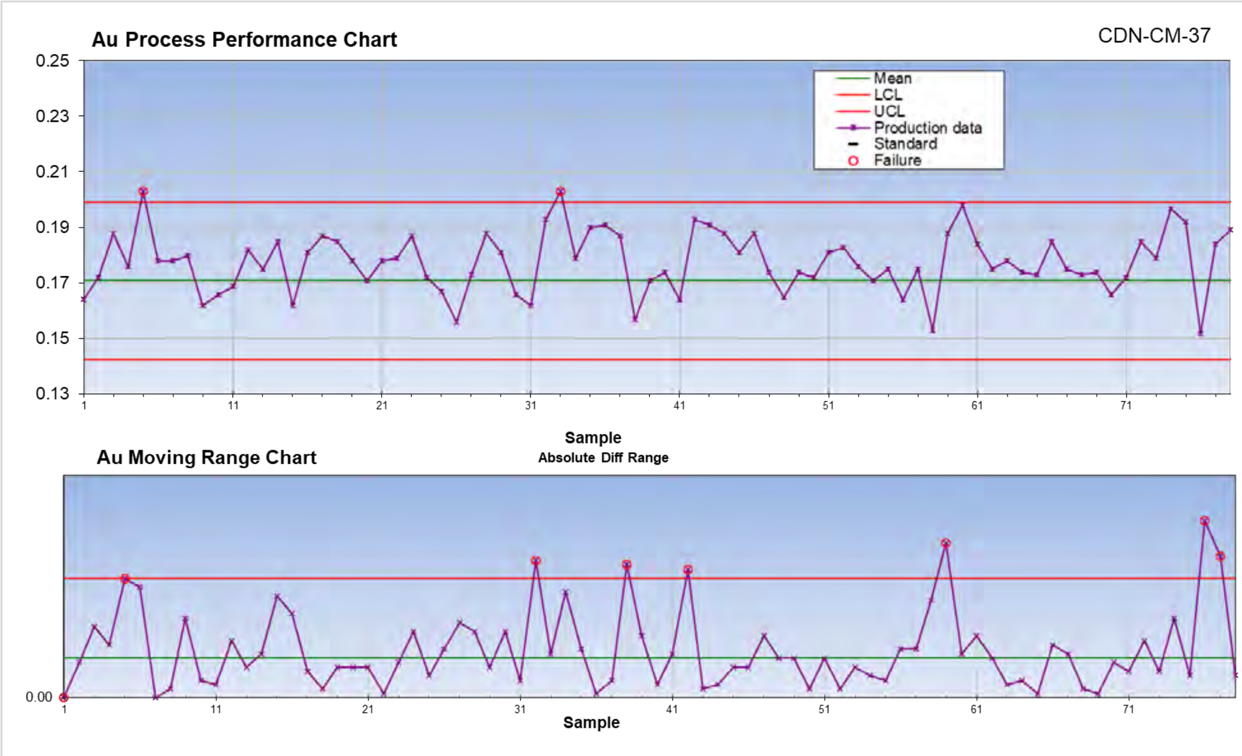


Figure 11-16: Toroparu Deposit Standard CDN-CM-37 gold (g/t)

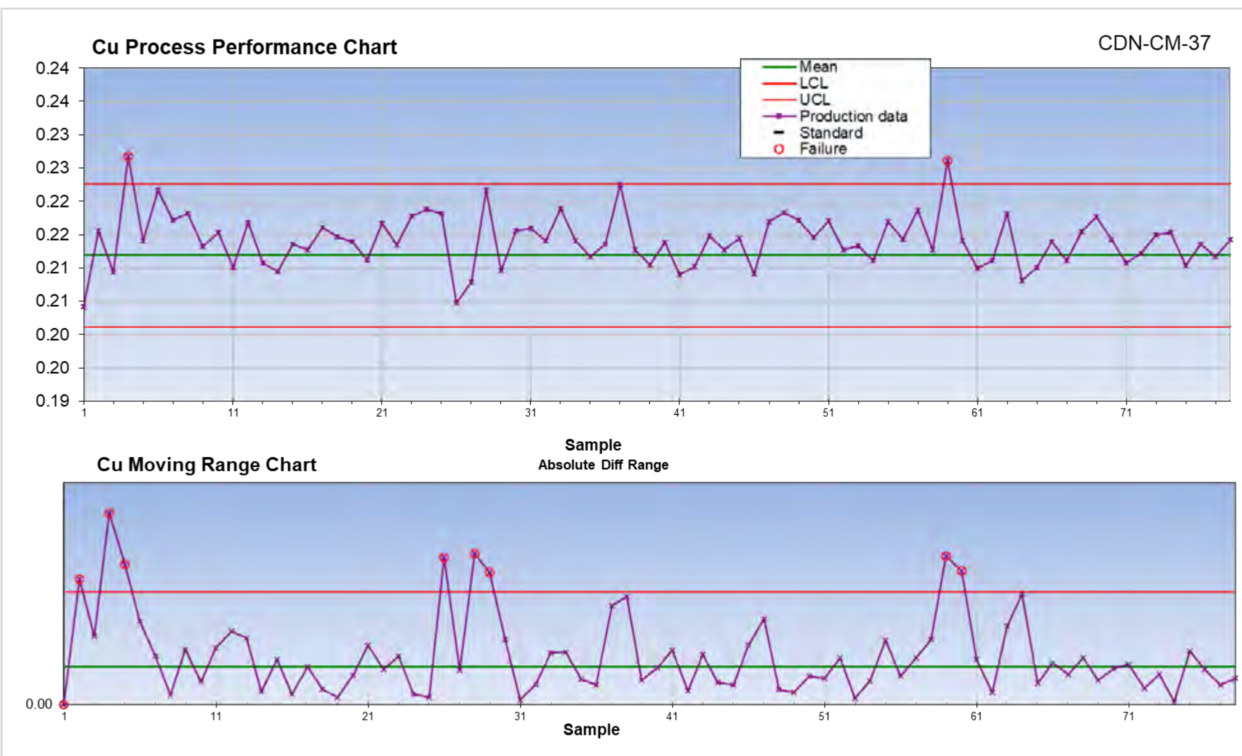


Figure 11-17: Toroparu Deposit Standard CDN-CM-37 copper (%)

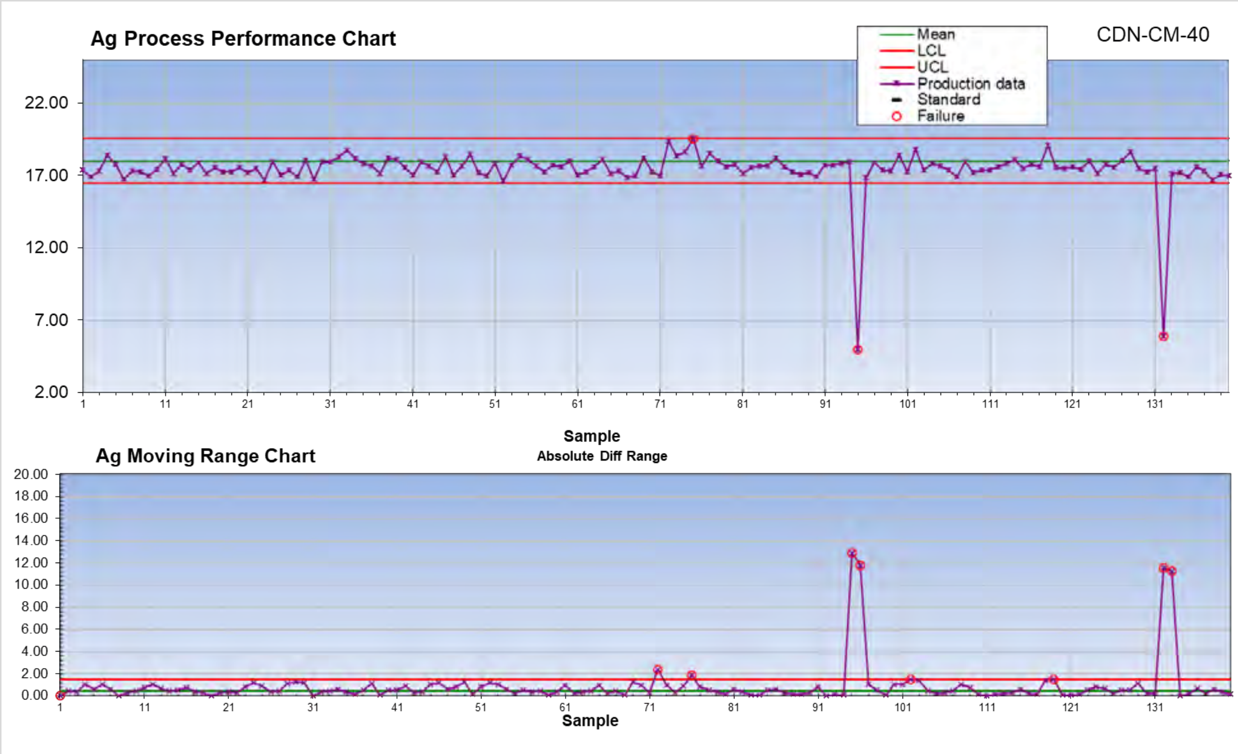


Figure 11-18: Toroparu Deposit Standard CDN-CM-40 silver (g/t)

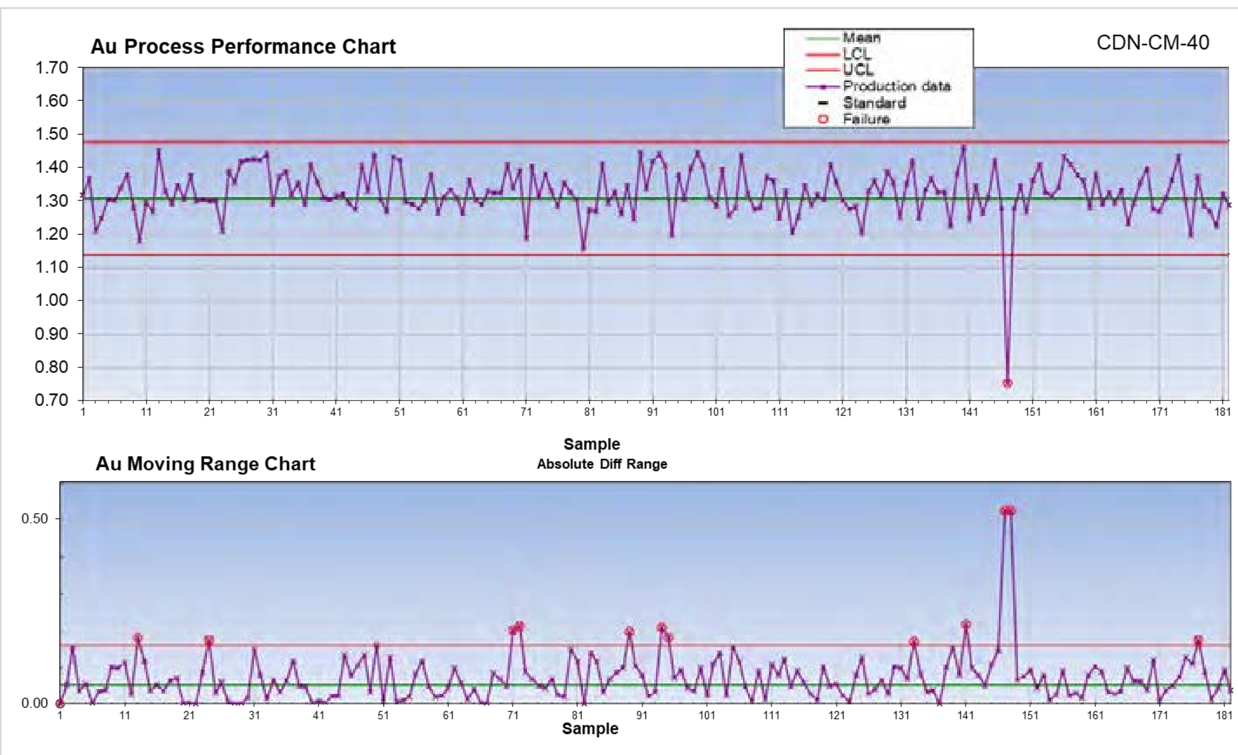


Figure 11-19: Toroparu Deposit Standard CDN-CM-40 gold (g/t)

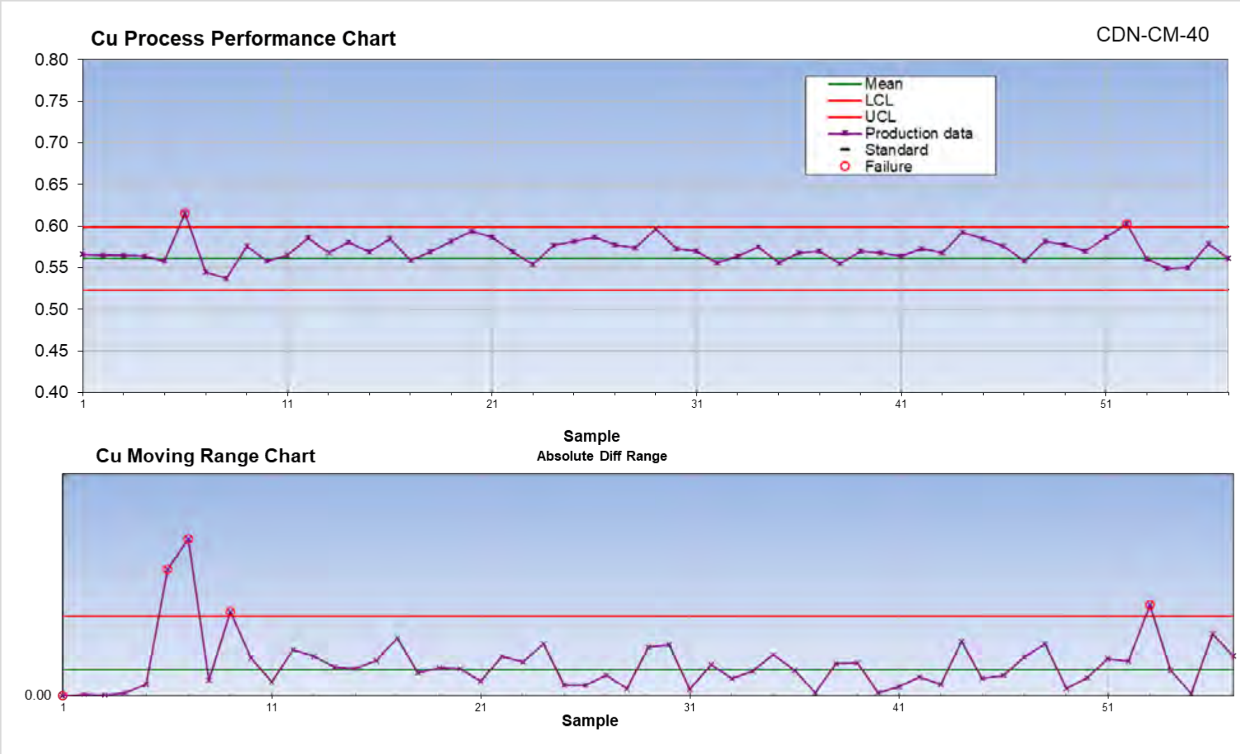


Figure 11-20: Toroparu Deposit Standard CDN-CM-40 copper (%)

Blanks

The Company submitted 854 coarse blanks for 2020 to 2021 as part of its QA/QC process for silver, gold, and copper (Figure 11-21, Figure 11-22 and Figure 11-23). The coarse blanks for silver and gold show few variances and few outliers. The blank values for copper show high variance and no outliers. There is no obvious correlation between the blank values and those samples immediately preceding. No significant contamination is evident.

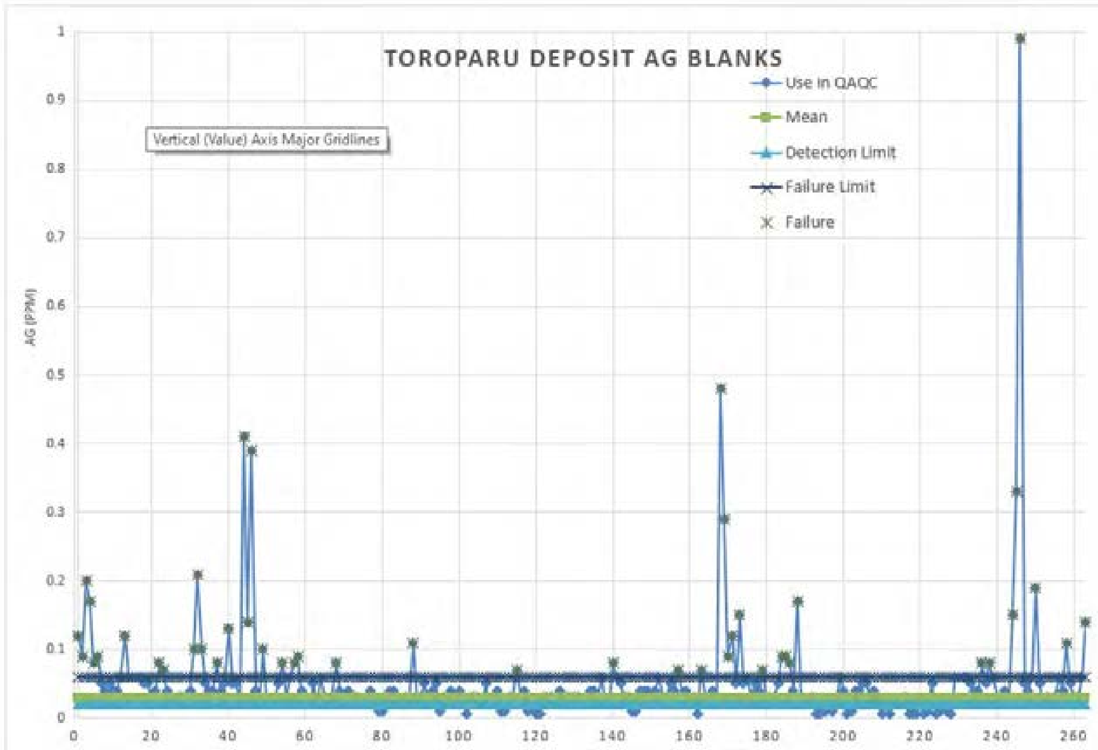


Figure 11-21: Toroparu Deposit coarse blanks for silver (g/t)

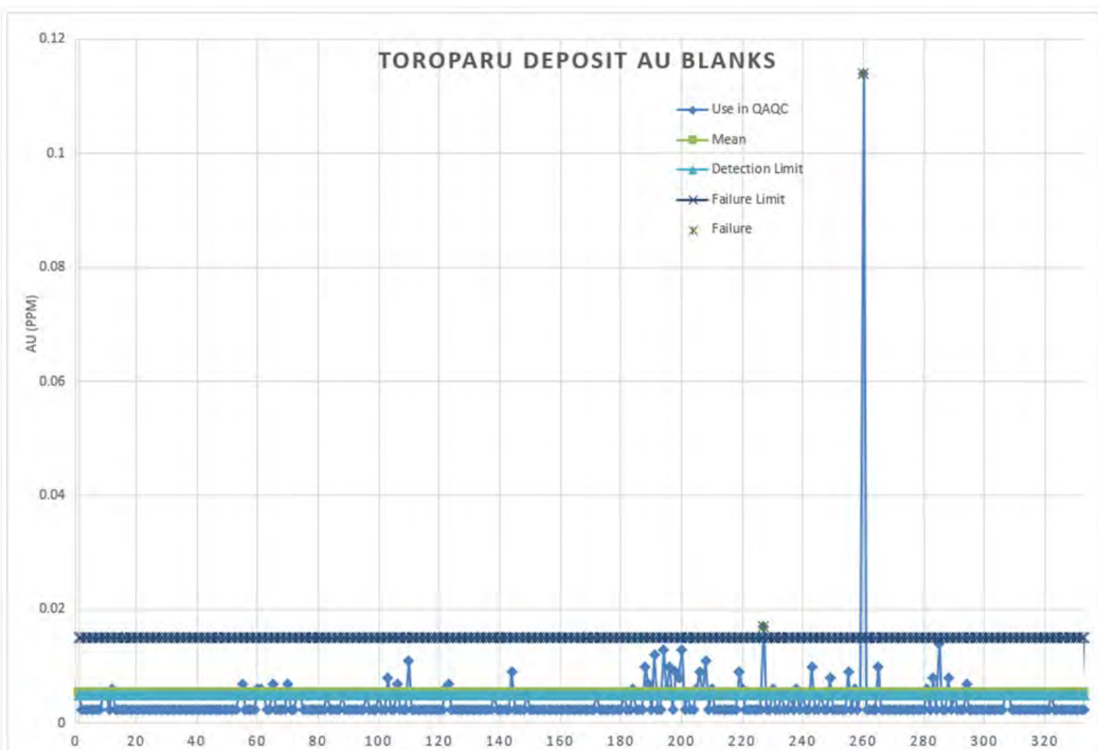


Figure 11-22: Toroparu Deposit coarse blanks for gold (g/t)

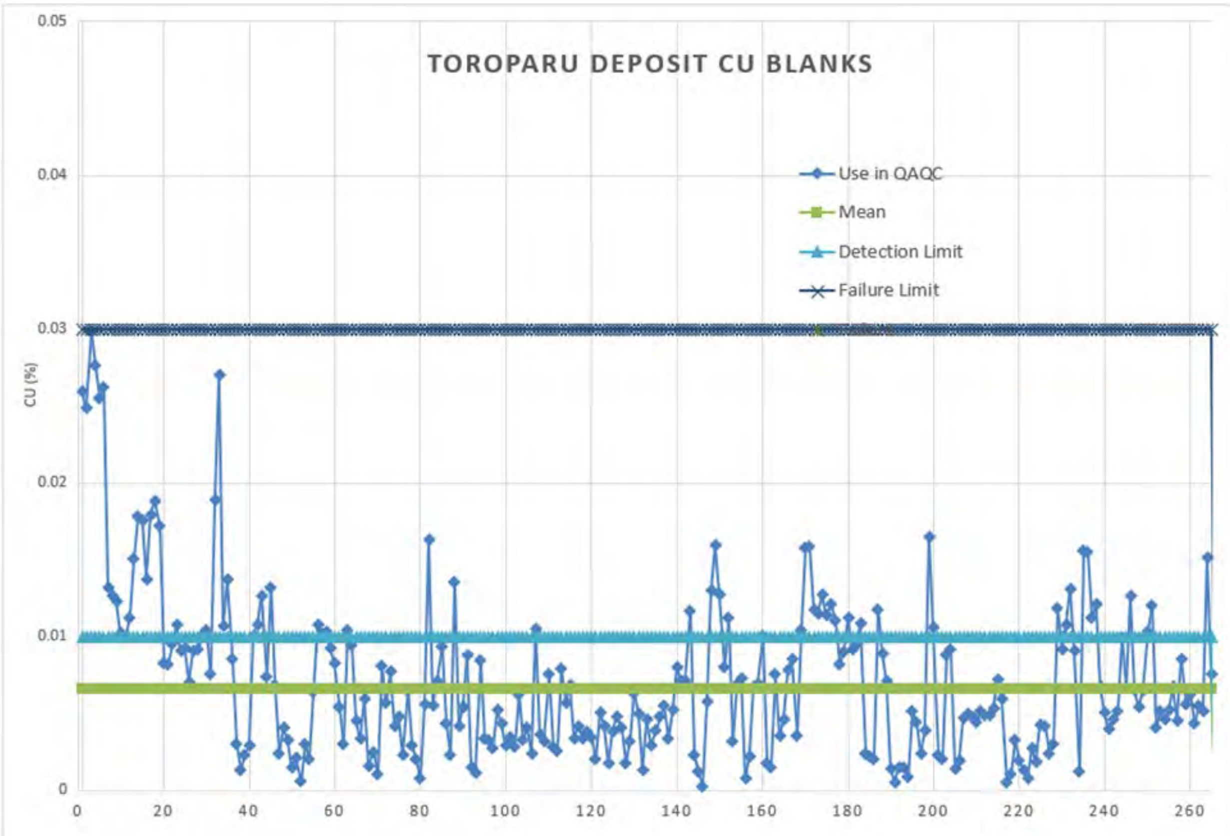


Figure 11-23: Toroparu Deposit coarse blanks for copper (%)

Laboratory Duplicates

The lab submitted 1,759 pulp and 713 crush duplicates and as part of their QA/QC process from 2020-2021. Both lab pulp and crush duplicates show excellent agreement for gold and copper with higher variance for silver (Figure 11-24, Figure 11-25, Figure 11-26, Figure 11-27, Figure 11-28 and Figure 11-29).

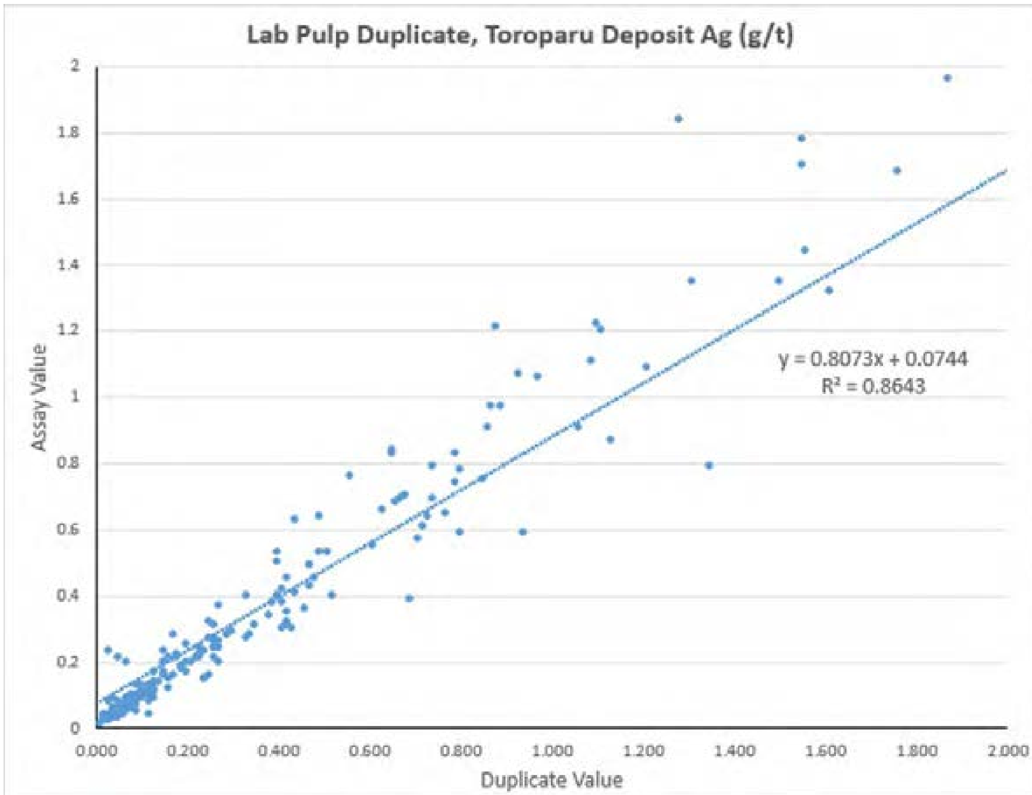


Figure 11-24: Toroparu Deposit lab pulp duplicates for silver (g/t)

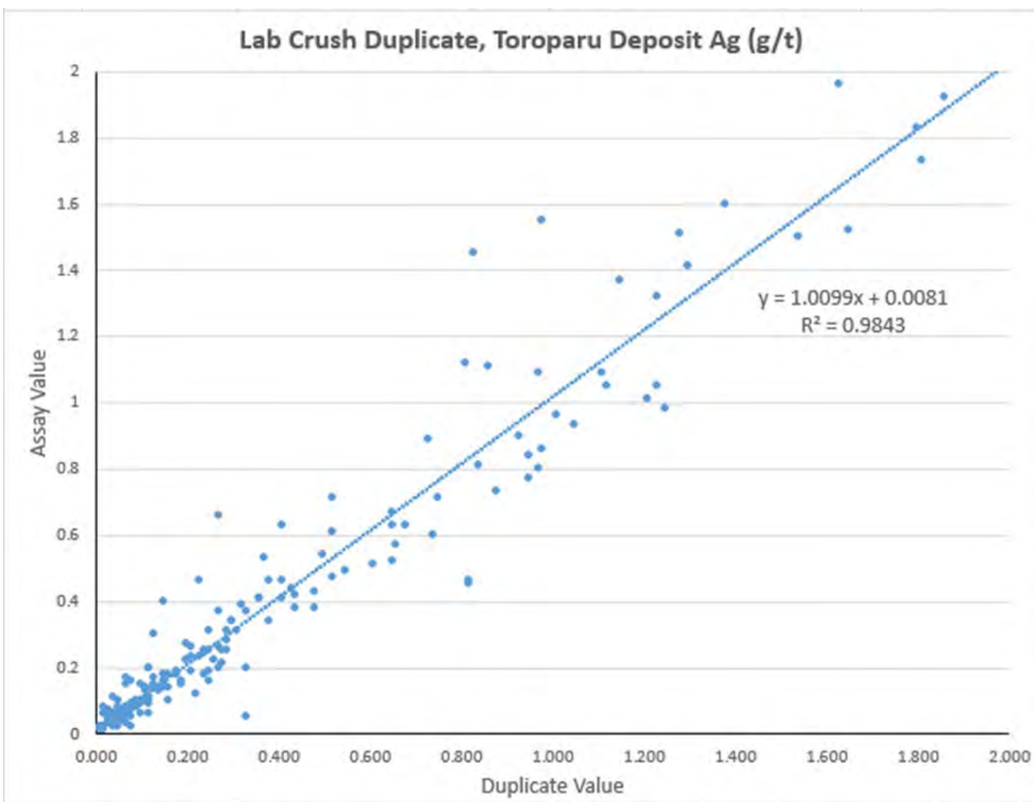


Figure 11-25: Toroparu Deposit lab crush duplicates for silver (g/t)

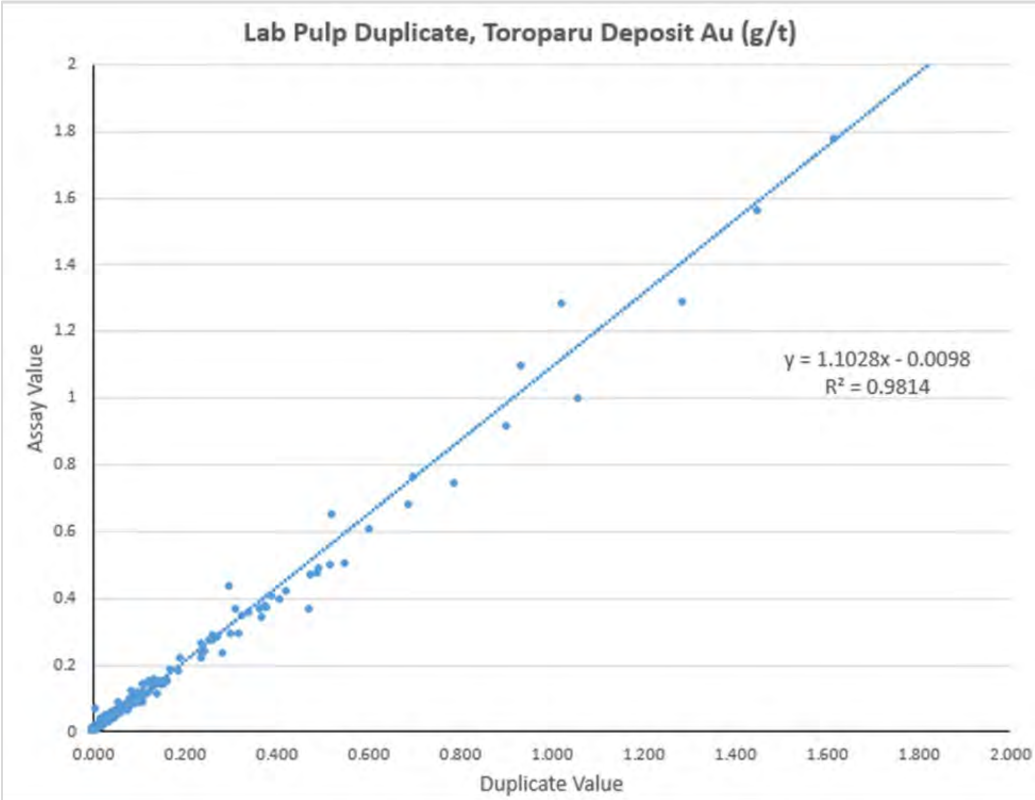


Figure 11-26: Toroparu Deposit lab pulp duplicates for gold (g/t)

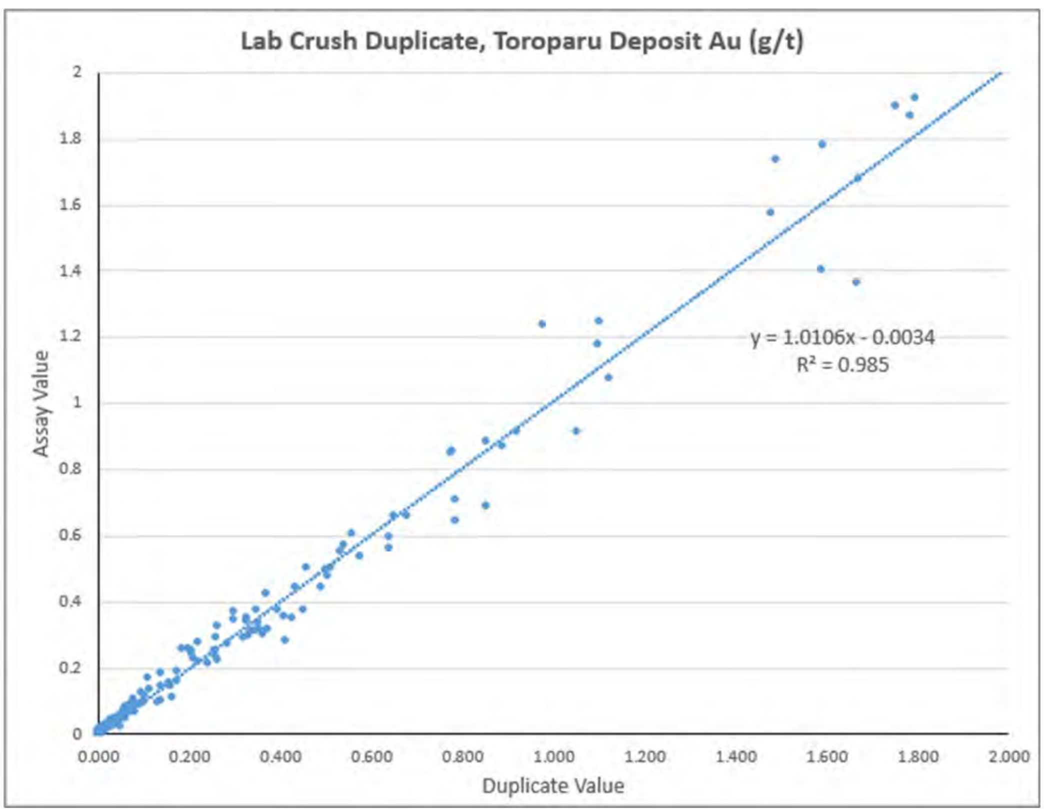


Figure 11-27: Toroparu Deposit lab crush duplicates for gold (g/t)

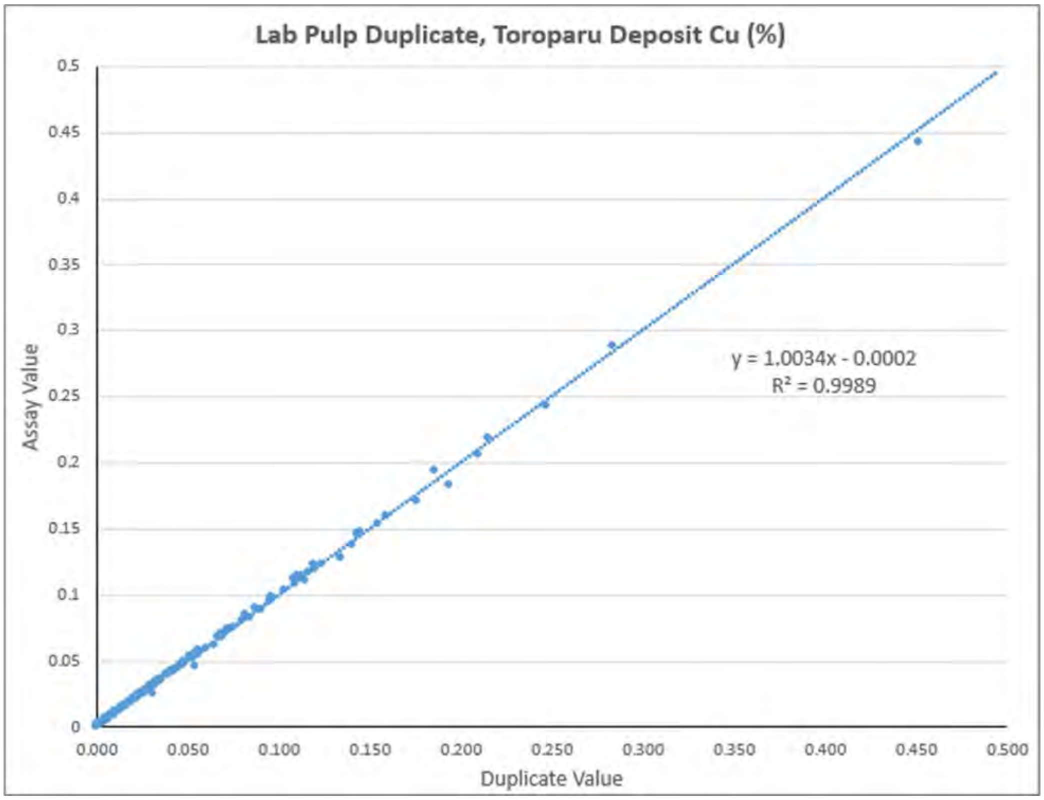


Figure 11-28: Toroparu Deposit lab pulp duplicates for copper (%)

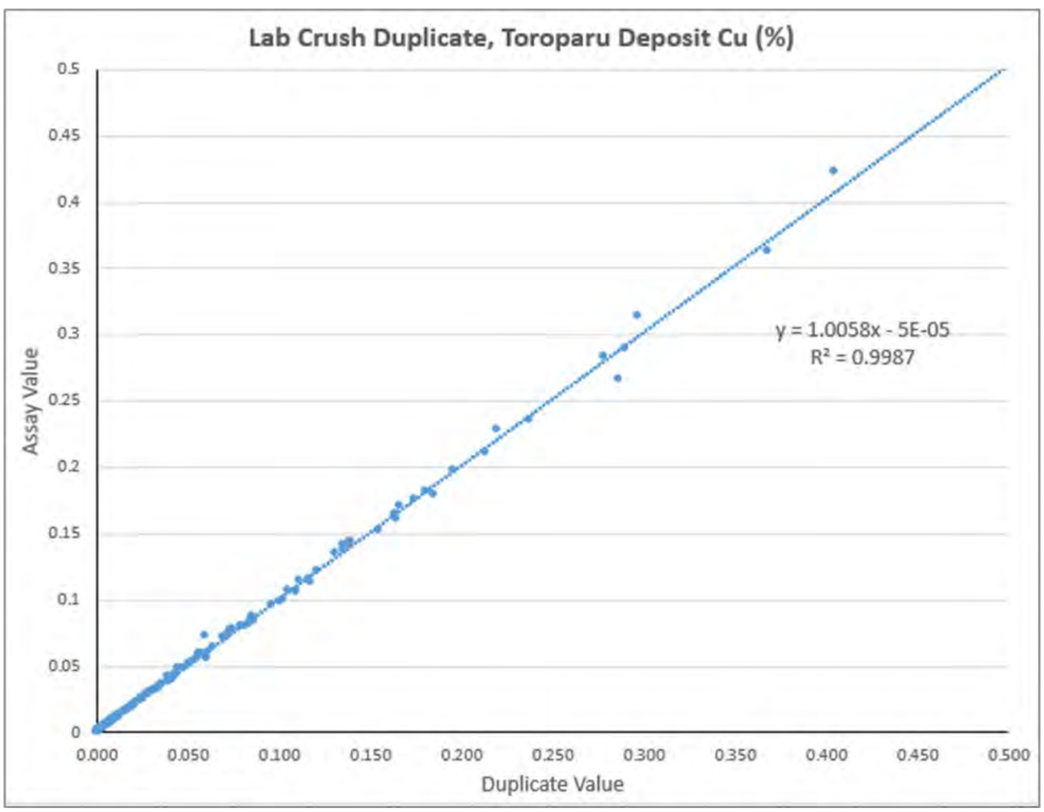


Figure 11-29: Toroparu Deposit lab crush duplicates for copper (%)

11.4.3 Sona Hill Deposit 2012, 2017-2018

Standards

The Company submitted six different CRMs as part of its QA/QC process with a total of 421 CRM during 2012, 2017-2018 (Table 11-3). The standards performed well, with few outliers being exhibited; these were mainly confined to CDN-CM-12, CDN-CM-13, and CDN-CM-14. Figure 11-30 through Figure 11-36 demonstrate the performance ranges of the various CRM sample types.

Table 11-3: Sona Hill Deposit 2012, 2017-2018 CRM Result Summary

Standard	Count	Best Value Au (g/t)	Mean Value Au (g/t)	Bias (%)
CDN-CGS-29	37	0.228	0.28	0.052
CDN-CM-12	93	0.686	0.715	0.029
CDN-CM-13	71	0.74	0.763	0.023
CDN-CM-14	90	0.792	0.813	0.021
CDN-CM-15	83	1.253	1.283	0.03
CDN-CM-19	47	2.11	2.135	0.025

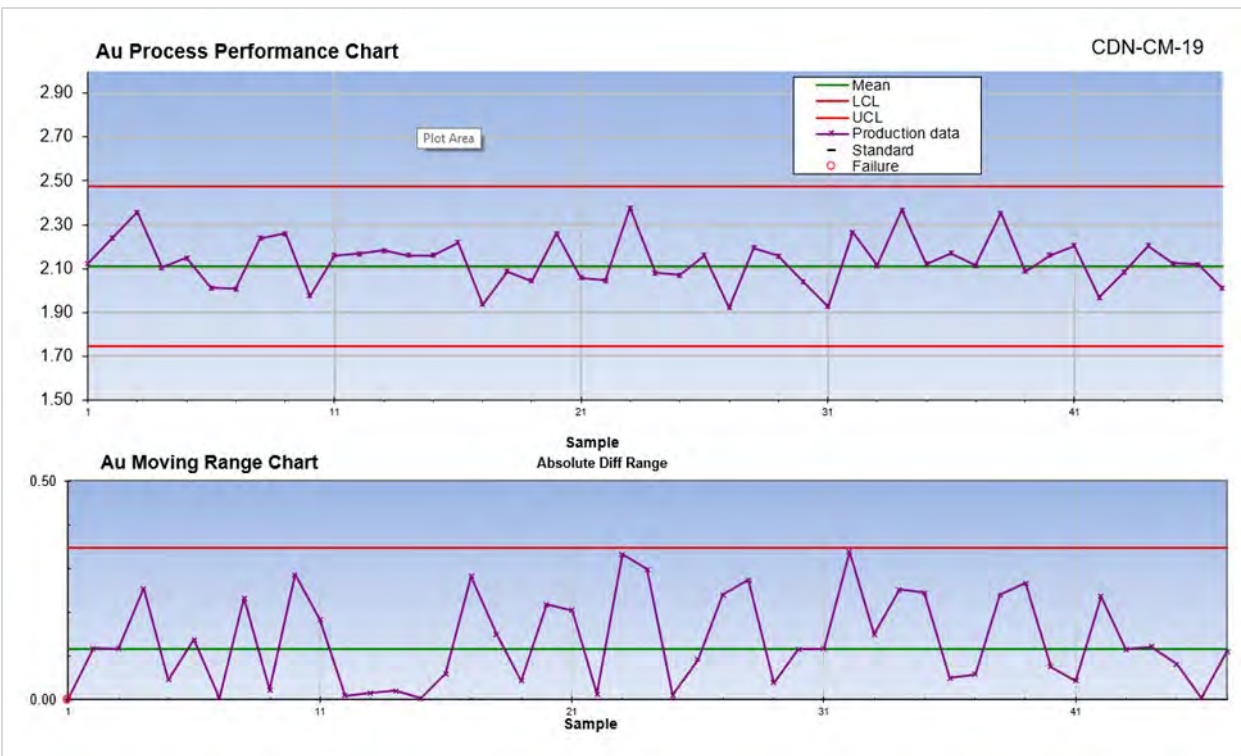


Figure 11-30: Toroparu and Sona Hill Deposits Standard CDN-CM-19 gold (g/t)

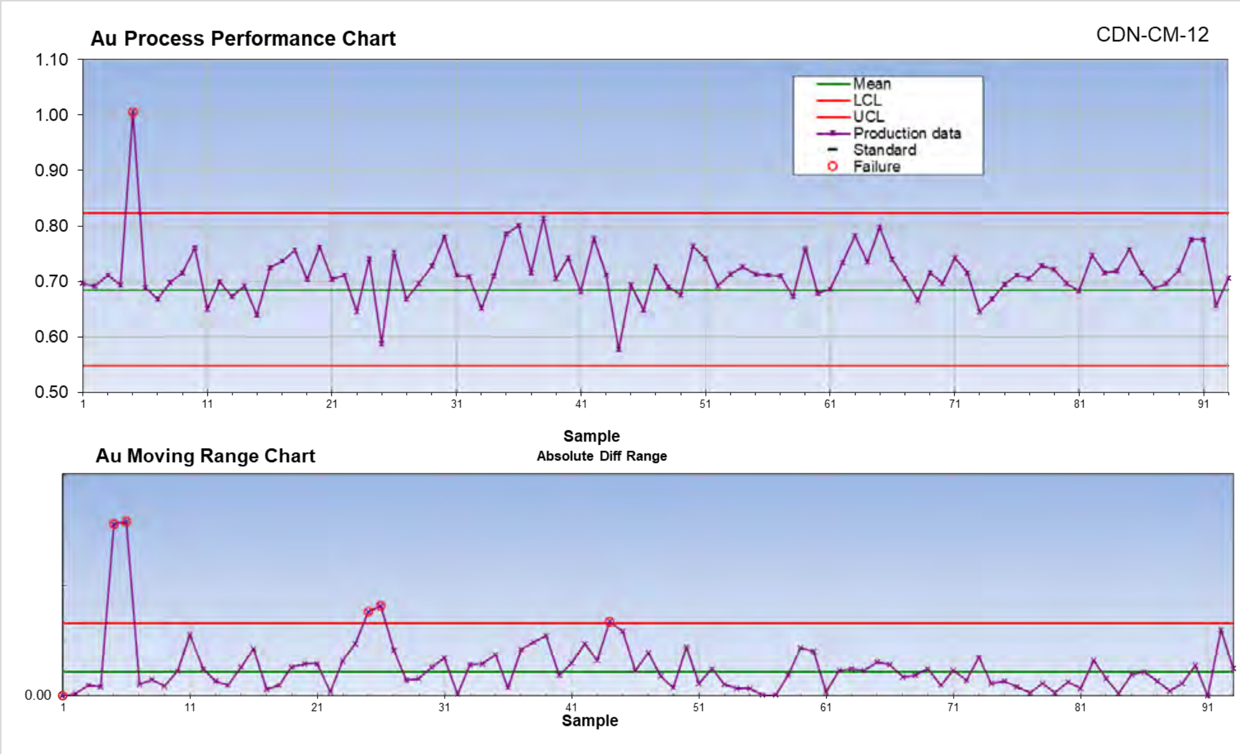


Figure 11-31: Toroparu and Sona Hill Deposits Standard CDN-CM-12 gold (g/t)

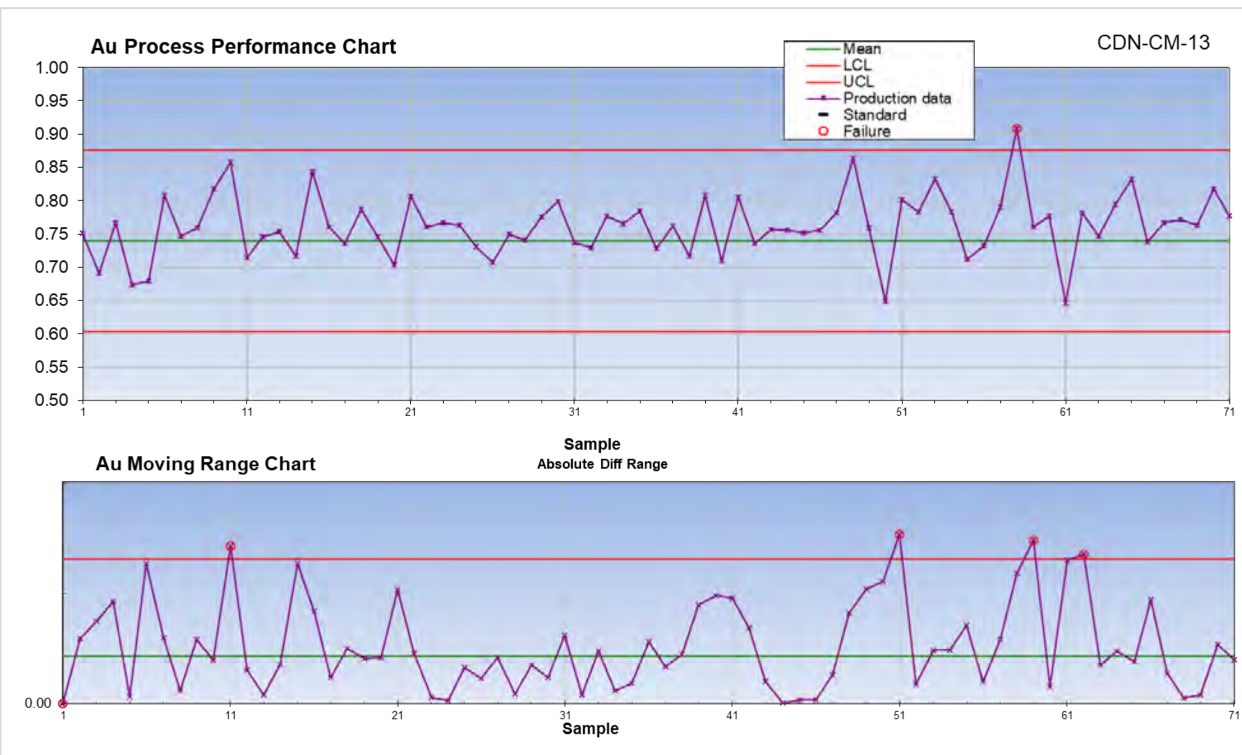


Figure 11-32: Toroparu and Sona Hill Deposit Standard CDN-CM-13 gold (g/t)

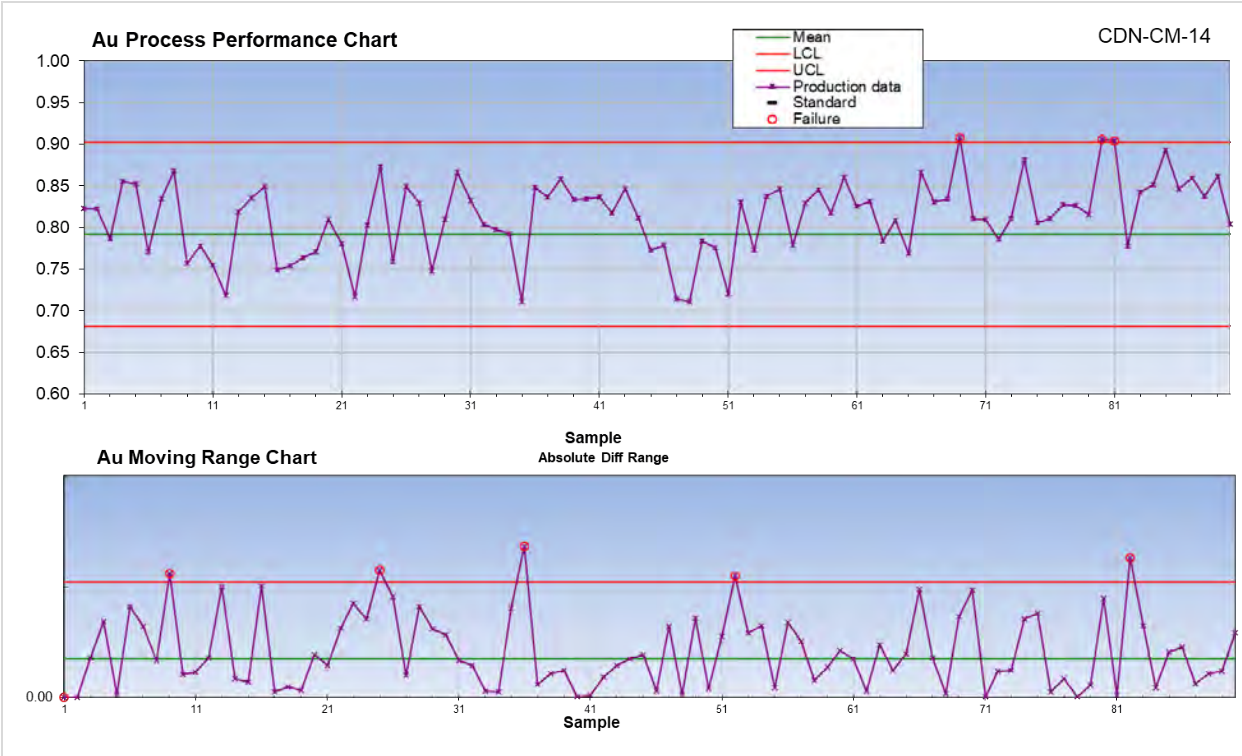


Figure 11-33: Toroparu and Sona Hill Deposit Standard CDN-CM-14 gold (g/t)

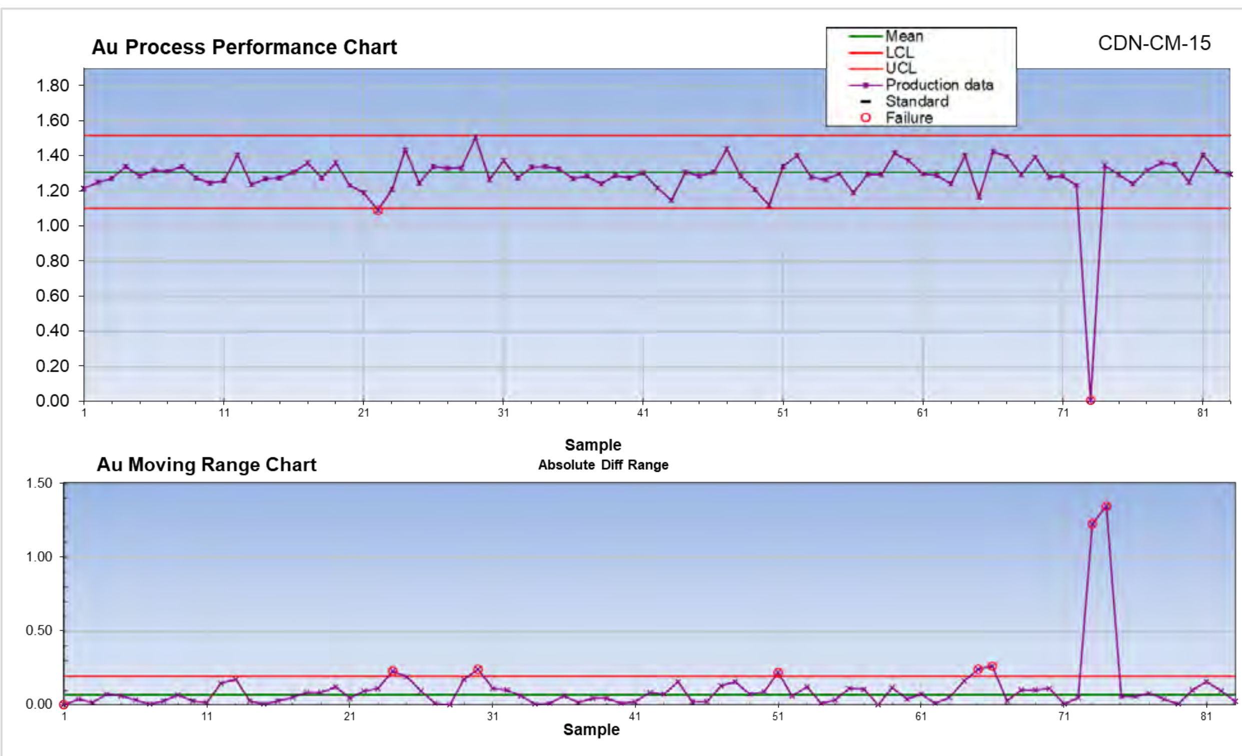


Figure 11-34: Toroparu and Sona Hill Deposits Standard CDN-CM-15 gold (g/t)

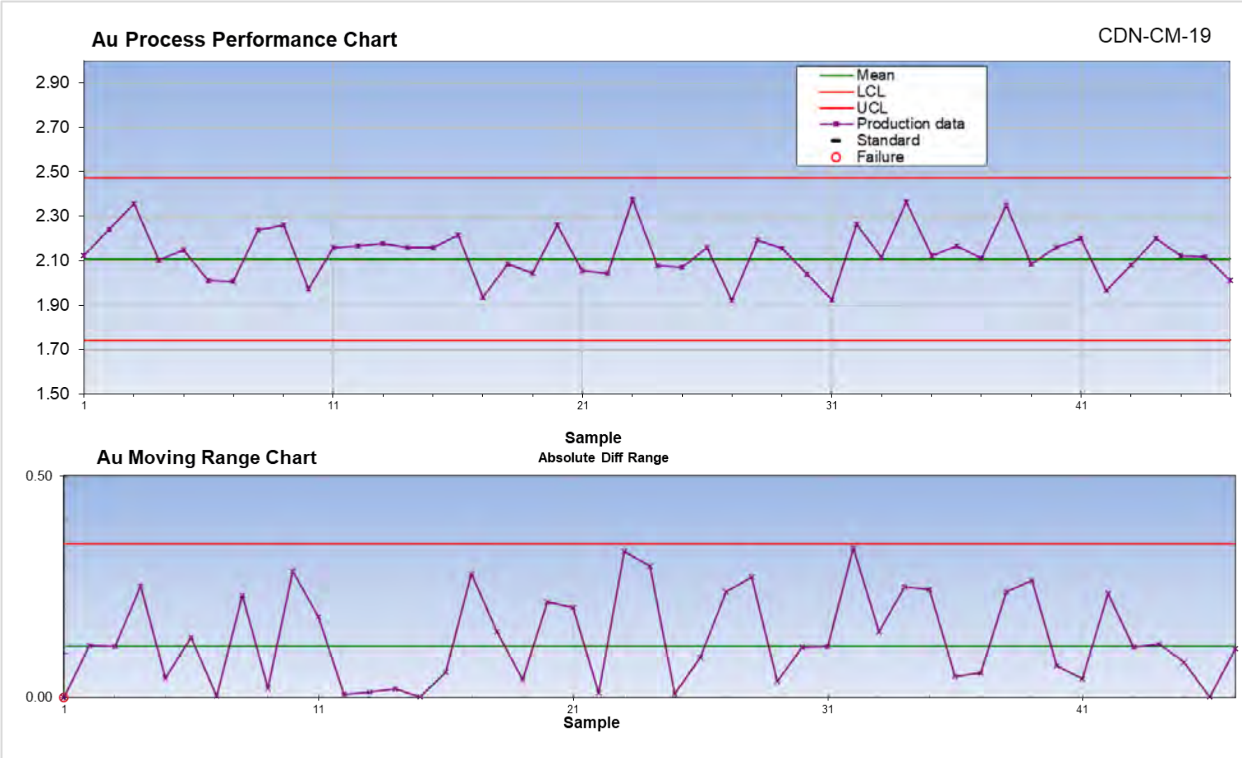


Figure 11-35: Toroparu and Sona Hill Deposits Standard CDN-CM-19 gold (g/t)

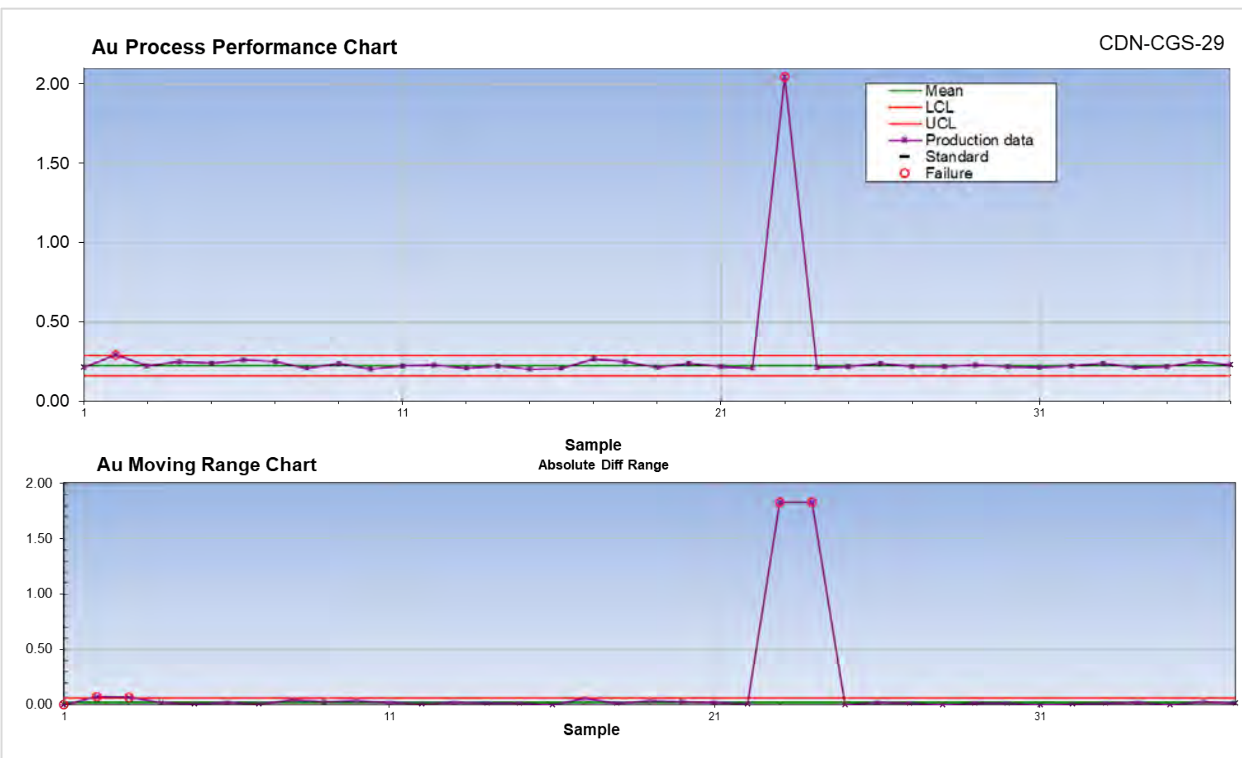


Figure 11-36: Toroparu and Sona Hill Deposits Standard CDN-CGS-29 gold (g/t)

Blanks

The Company submitted 216 coarse blanks for the Sona Hill Deposit during 2012, 2017, and 2018 as part of its QA/QC process. The coarse blanks for gold show some variance and outliers (Figure 11-37). There is no obvious correlation between the blank values and those samples immediately preceding. No significant carryover is evident.

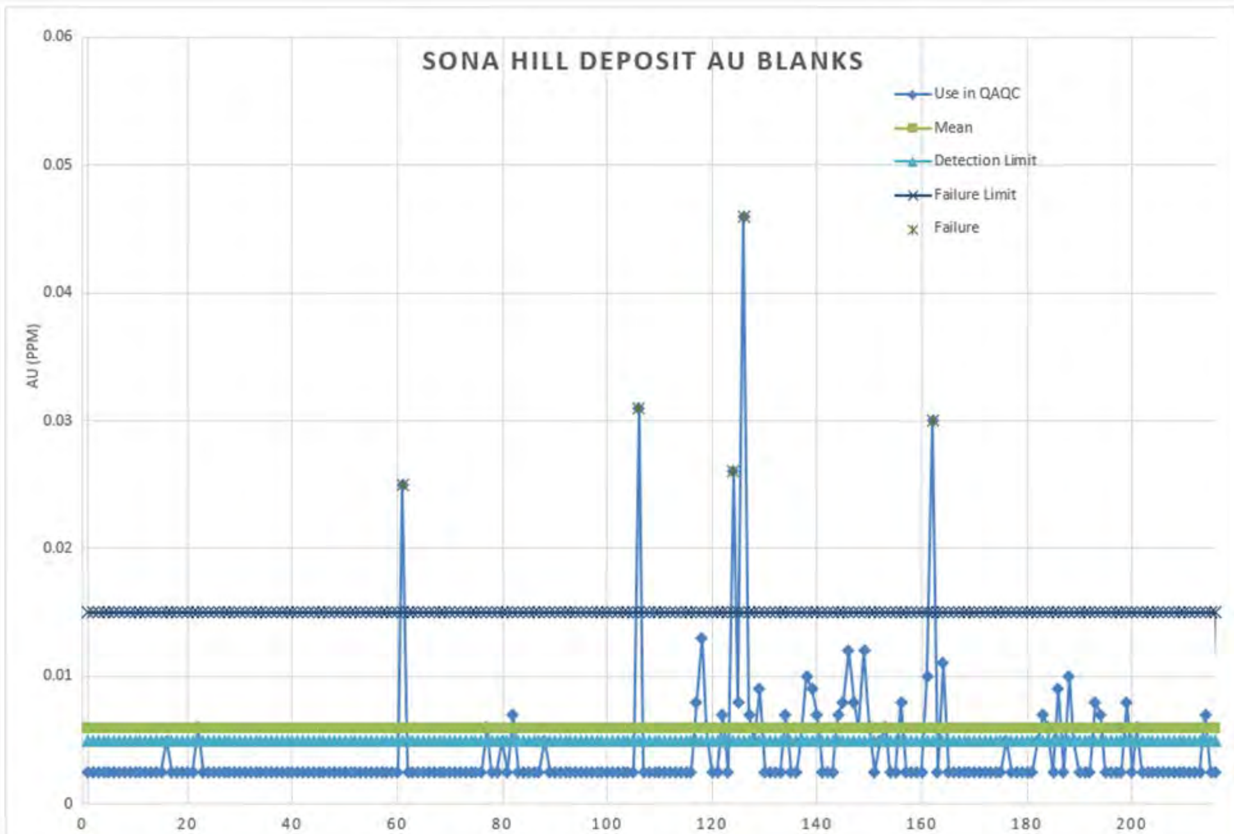


Figure 11-37: Toroparu and Sona Hill Deposits coarse blanks for gold (g/t)

Field and Laboratory Duplicates

The Company submitted 257 field duplicates, and the lab submitted 217 pulp lab duplicates and 180 crush lab duplicates as part of their QA/QC process from 2012, 2017-2018. The field duplicates demonstrate good agreement for gold (Figure 11-38). Lab run crush and pulp duplicates also demonstrate good agreement (Figure 11-39 and Figure 11-40).

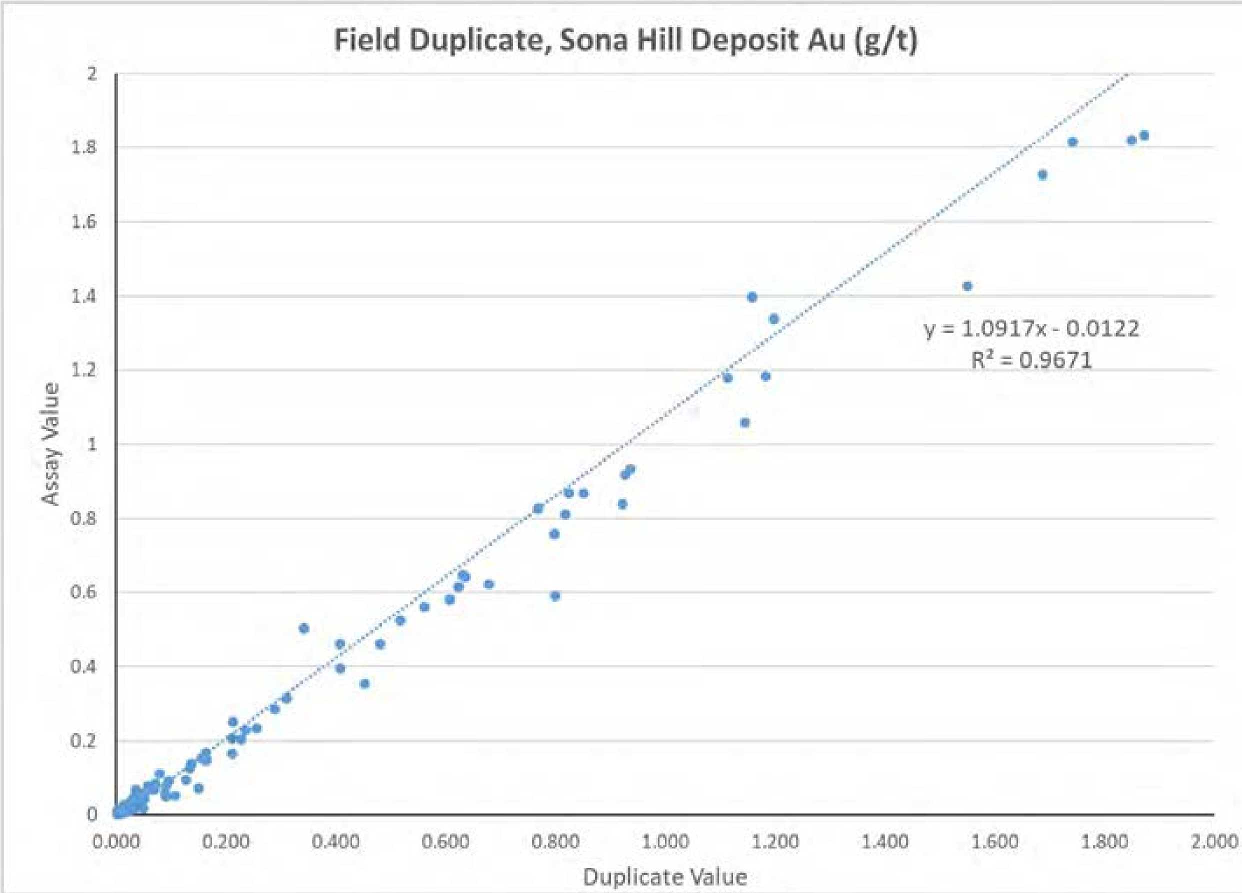


Figure 11-38: Sona Hill Deposit field duplicate for gold (g/t)

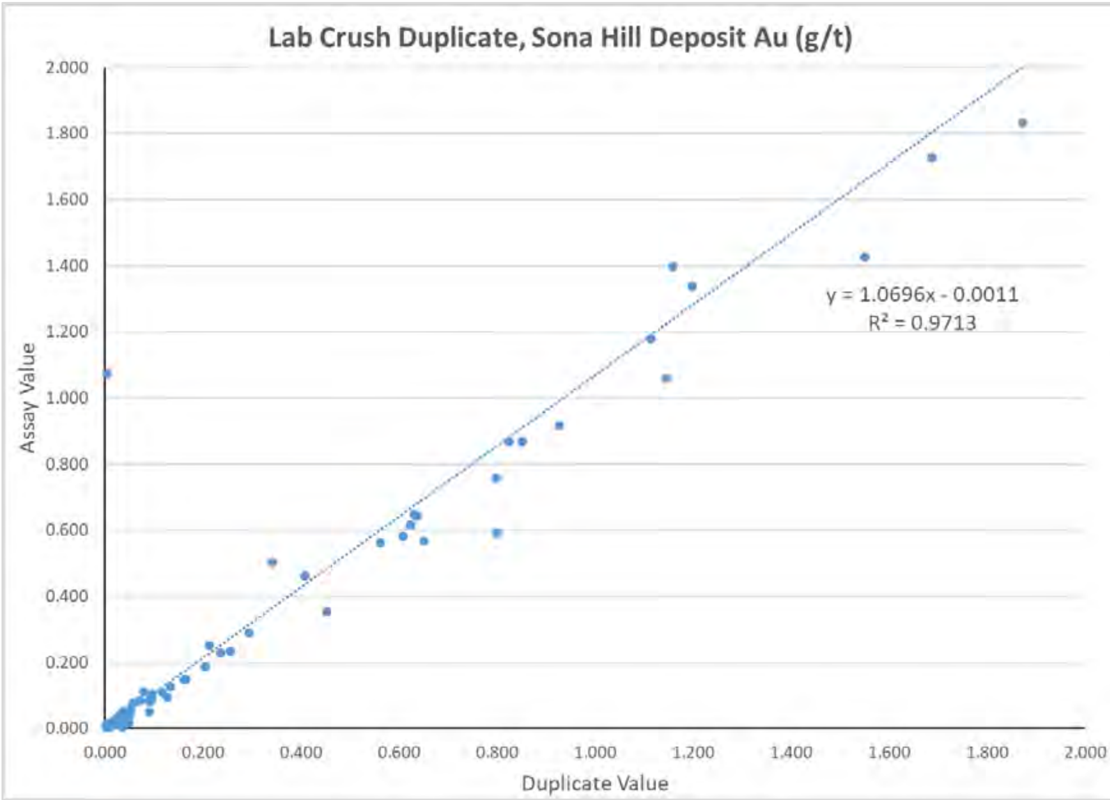


Figure 11-39: Sona Hill Deposit lab crush duplicates for gold (g/t)

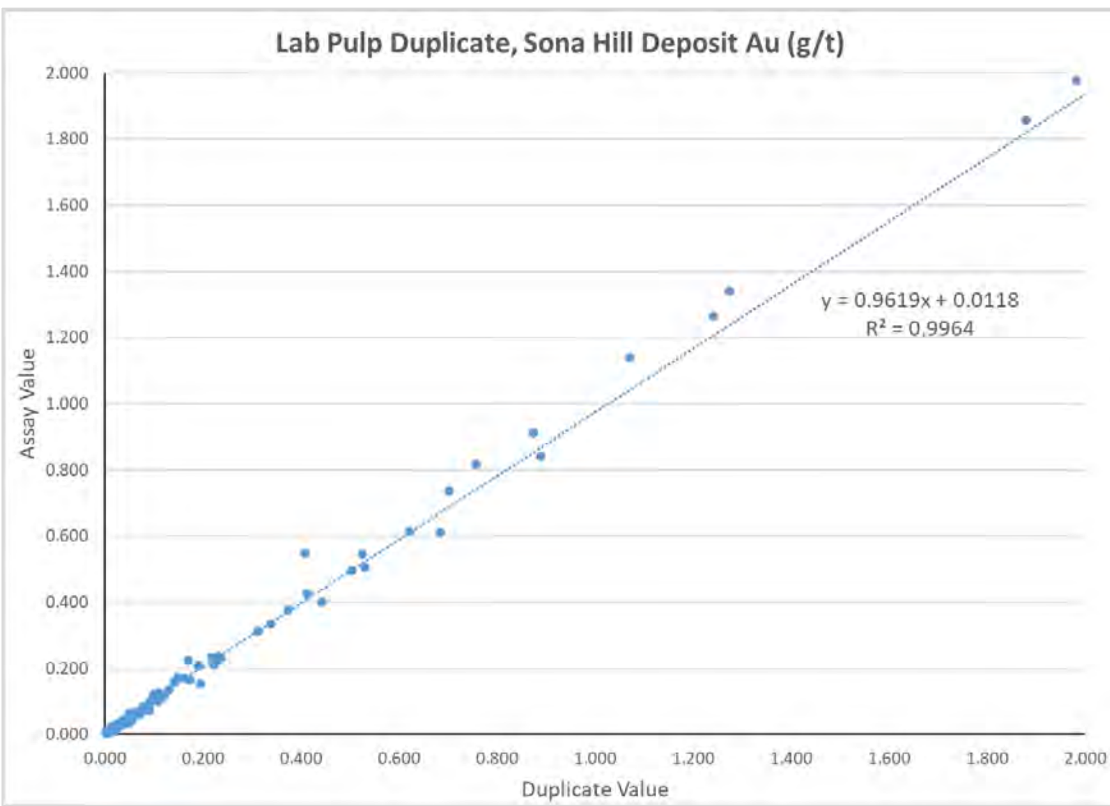


Figure 11-40: Sona Hill Deposit lab pulp duplicates for gold (g/t)

11.5 Security and Storage

The Company manages the security and is in possession of the drill core throughout logging, sampling, and delivery to the on site ACME facility. The facility is a large industrial wood building with wooden core racks secured with locks (Figure 11-41).

Figure 11-41: Secure core storage (A) and secure wooden core racks (B), inside storage.

11.6 Qualified Person's Opinion on the Adequacy of Sample Preparation, Security, and Analytical Procedures.

Nordmin has been supplied with all raw QA/QC data and has reviewed and completed an independent check of the results for all the Project sampling programs. It is Nordmin's opinion that the sample preparation, security, and analytical procedures used by all parties are consistent with standard industry practices and that the data is suitable for the 2021 Mineral Resource Estimate. Nordmin identified further recommendations to the Company to ensure the continuation of a robust QA/QC program but has noted no material concerns with the geological or analytical procedures used or the quality of the resulting data.

12 DATA VERIFICATION

Nordmin completed several data validation checks throughout the duration of the 2021 Mineral Resource Estimate. The verification process included a site visit to the Project by the QP to review surface geology, drill core geology, geological procedures, chain of custody of drill core, sample pulps, lab audit, and the collection of independent samples for metal verification. The data verification included:

- A survey spot check of drill collars.
- Historical mine workings.
- A spot check comparison of assays from the drill hole database against original assay records (lab certificates).
- A spot check of drill core lithologies recorded in the database versus the core located in the core storage shed.
- A review of the QA/QC performance of the drill programs.

Nordmin has also completed additional data analysis and validation, as outlined in Section 11.

12.1 Nordmin Site Visit 2021

A site visit to the Project was carried out between April 24 and April 25, 2021, by Glen Kuntz, P.Geo., QP for Mineral Resources. Greg Barnes – Executive VP accompanied Mr. Kuntz, and Project geologists Bjorn Jeune and Amber Markan, who collectively have been involved with the Project for multiple years. Activities during the site visit included the:

- Review of the geological and geographical setting of the Toroparu Deposit (Main, NW, and SE Areas), and the Sona Hill Deposit.
- Review and inspection of the site geology, low, med, and high-grade mineralization, and structural controls concerning gold, copper, and silver distribution.
- Review of the drilling, logging, sampling, analytical and QA/QC procedures.
- Review of the chain of custody of samples from the field to the assay lab.
- Review of the drill logs, drill core, storage facilities, and independent assay verification on selected core samples.
- Confirmation of a variety of drill hole collar locations.
- Review of the structural measurements recorded within various drill logs and how they are utilized within the Company's geological/structural model.
- Validation of a portion of the drill hole database.
- Review of the artisanal miners' areas of disturbance.

The Company geologists completed the geological mapping, core logging, and sampling associated with the drill programs. Therefore, Nordmin used the Company's database to review the core logging procedures, the collection of samples, and the chain of custody associated with the drilling and sampling programs. The Company provided Nordmin with excerpts from the drill database (GEMS Logger) for the Project and electronic copies of the original logging and assay reports (Figure 12-1).

Figure 12-1: Reviewing drill core using the Company's drill logging program (GEMS Logger)

No significant issues were identified during the site visit. However, three suggestions that should be incorporated into the company's workflow include:

1. Standardization of operating procedures.
2. Regular detailed drill audit.
3. Insertion of a blank sample after all samples with noted VG or expected high-grade gold. (This procedure has been included for the 2021 drill programs).

The Company employs a rigorous QA/QC protocol, including the routine insertion of field duplicates, laboratory pulp duplicates, blanks, and certified reference standards. Nordmin was provided with an excerpt from the database for review.

The collection and use of the structural information using the Reflex IQ Logger were reliable and representative of the drilled structure features (Figure 12-2).

Figure 12-2: Reflex IQ Logger, collection of structural measurements

The geological data collection procedures (spoken and unwritten) and the chain of custody were consistent with industry standards and following the Company's internal procedural documentation. As a result, Nordmin was able to verify the quality of geological and sampling information and develop an interpretation of gold, copper, and silver grade distributions appropriate for the Mineral Resource Estimate.

12.1.1 Field Collar Validation

The QP confirmed the various 2020 and 2021 drill collar locations used within the Mineral Resource Estimate. Each drill collar that the Company drilled had a steel pipe or steel bars with weld marking of the drill hole name (Figure 12-3). In addition, some of the drill holes had casing within the hole. Nordmin reviewed the hole collars within the database compared to a handheld GPS and determined that the collar locations are within acceptable error limits. Overall, the work found that the checked collars were within the ± 2 m to 5 m accuracy of the handheld GPS units (Figure 12-4).

Figure 12-3: Drill collars: (A) Sona Hill Deposit, and (B) Toroparu Deposit, Main Area

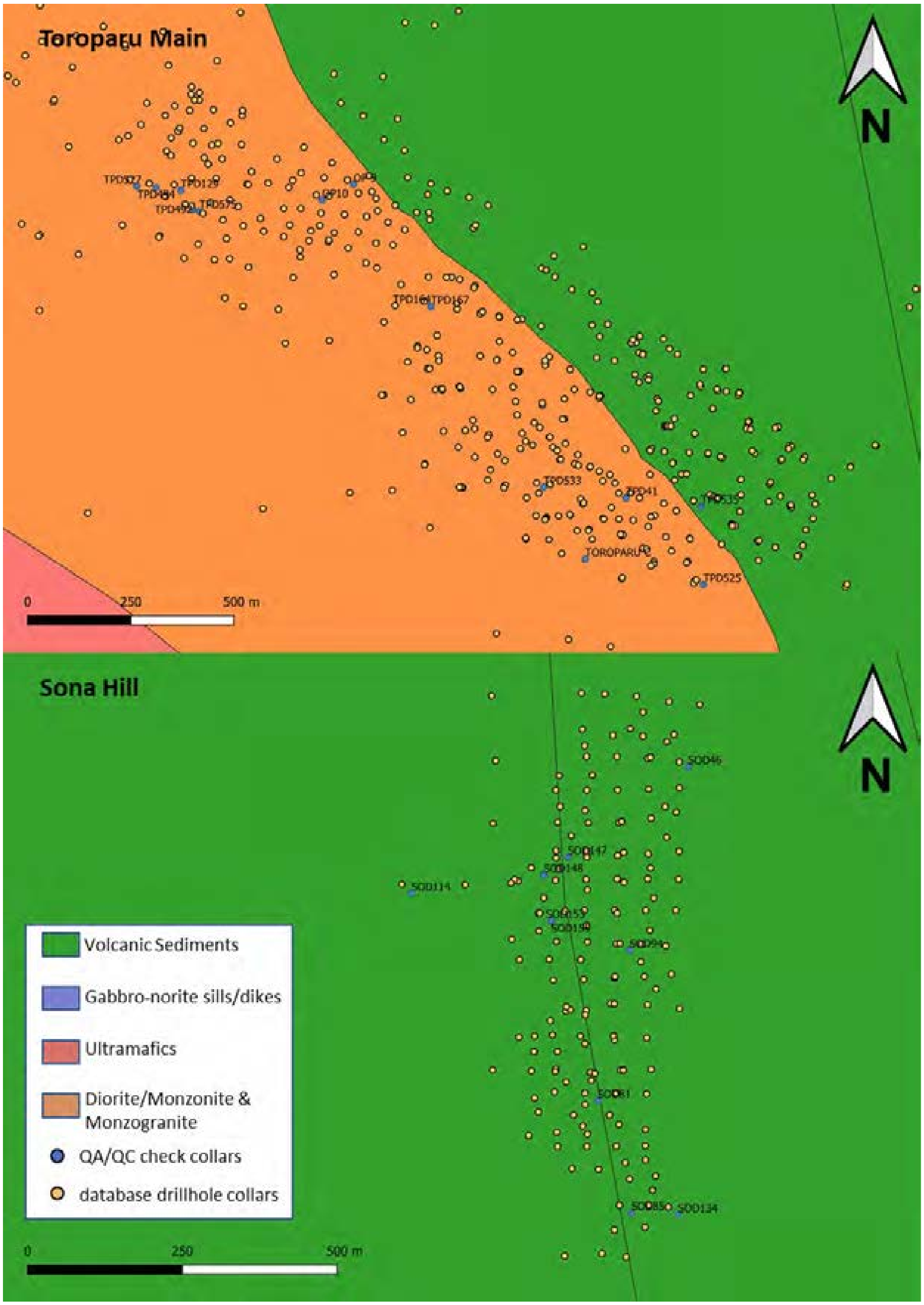


Figure 12-4: Map of database collar locations versus QA/QC check collar locations for the Toroparu Deposit, Main Area (top) and Sona Hill Deposit (bottom).

12.1.2 Core Logging, Sampling, and Storage Facilities

The Company drill holes were logged, photographed, and sampled on site at the core logging facility. The core is stored on site in one main location at the core storage facility (Figure 12-5, Figure 12-6 and Figure 12-7). In addition, coarse rejects that have not been consumed for geochemical analysis, as well as all pulps, are archived in the Company's secure storage facility on site.

Figure 12-5: Company logging and sampling facility

Figure 12-6: Company core cutting facility

Figure 12-7: Company core storage facility

12.1.3 2020 Check Assay Program

The QP selected various lower and higher-grade intervals from multiple Company drill holes for a total of 117 original 1.0 m to 1.5 m long intercepts. The purpose of this program was to check if the 1.5 m long original intervals were representative of the actual thickness of the higher-grade mineralization or if the actual thickness of the mineralization was significantly less. The Company quartered the core lengths and broke the samples into approximately 0.50 m sample lengths instead of the original 1.0 m to 1.5 m sample lengths for a total of 319 check samples. Figure 12-8 and Figure 12-9 show the grade distribution in the original sample intervals and the re-assayed check sample intervals, respectively.

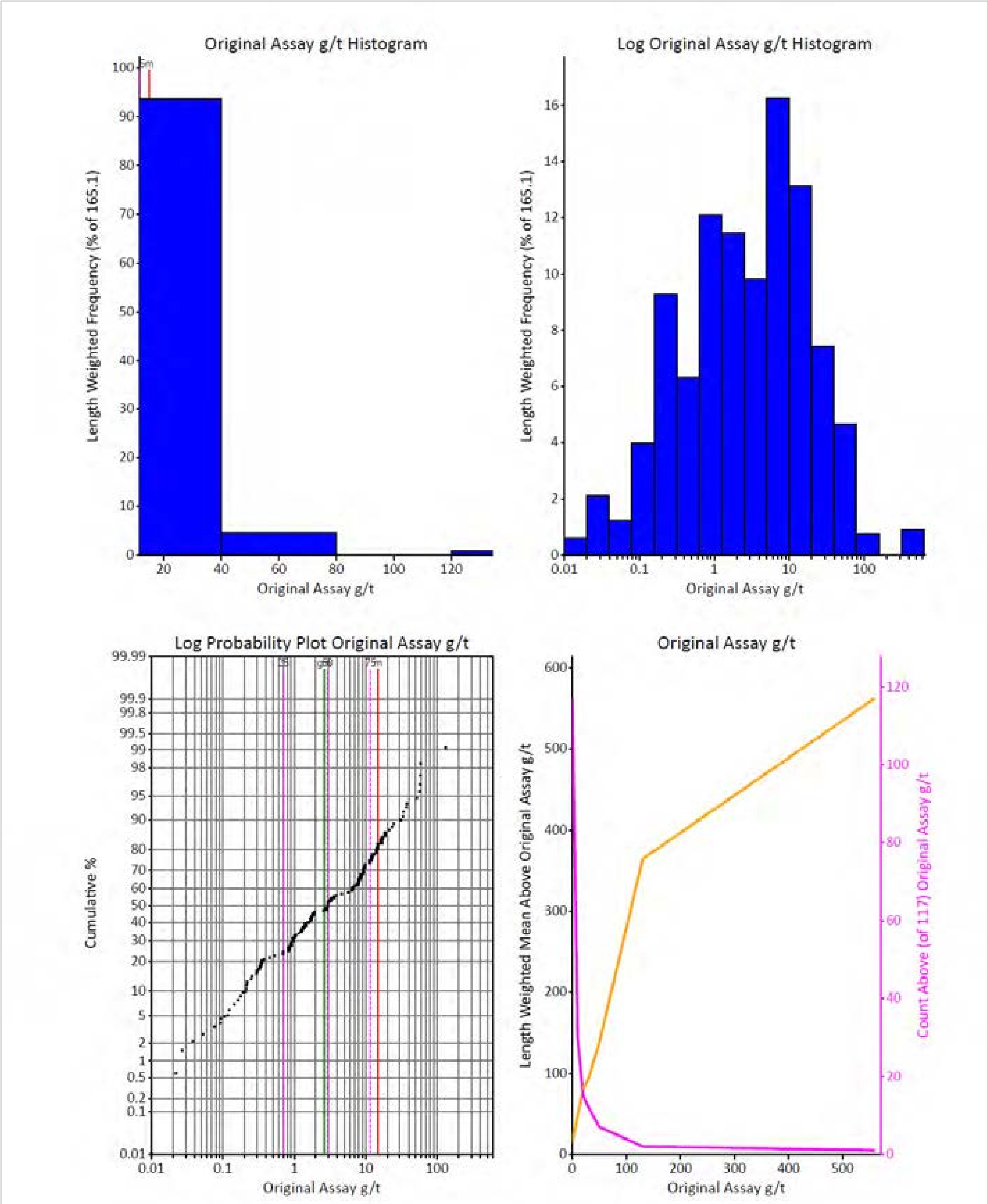


Figure 12-8 Original assay distribution.

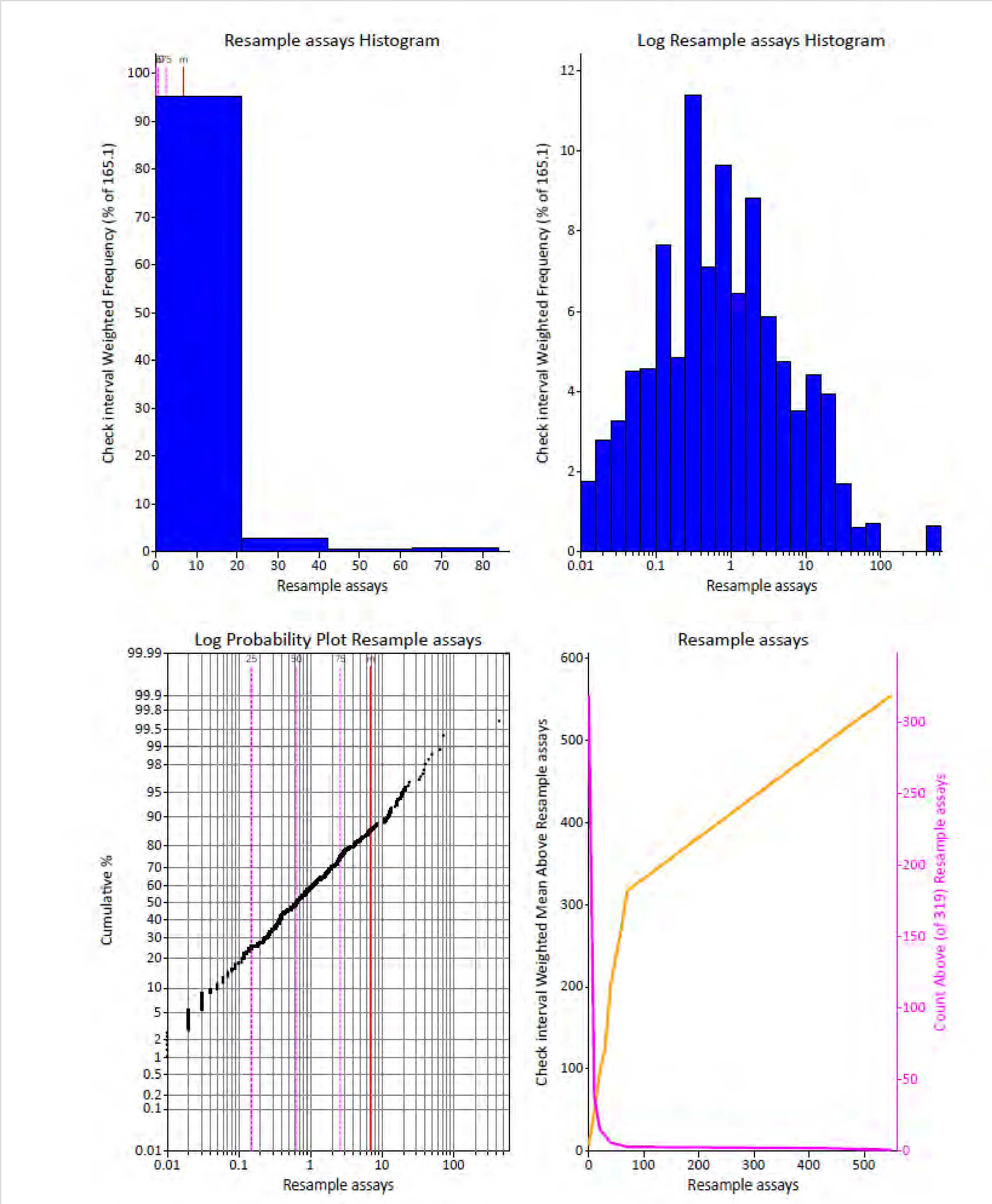


Figure 12-9 Re-assay sample distribution with varying sample lengths.

The comparison between the two sample groups does show a slightly different distribution. The shorter sample lengths have a higher CV compared to the longer sample lengths. The overall mean grade is lower with the re-assay samples compared to the original. Nordmin recommends that the Company continue to create sample lengths based on lithological, structural, and mineralization features rather than a standard 1.5 m sample distribution to allow the geologists to continue monitoring the gold, copper, and silver distribution within the lower grade and higher-grade structures.

However, when the grades are compared to the estimated block model gold grades, which use composited intervals, the bias between the initial sample grade and the re-sampled (variable length) samples is similar. Figure 12-10 to Figure 12-14 demonstrate the differences between the original assay, re-assay, and the estimated block model grade.

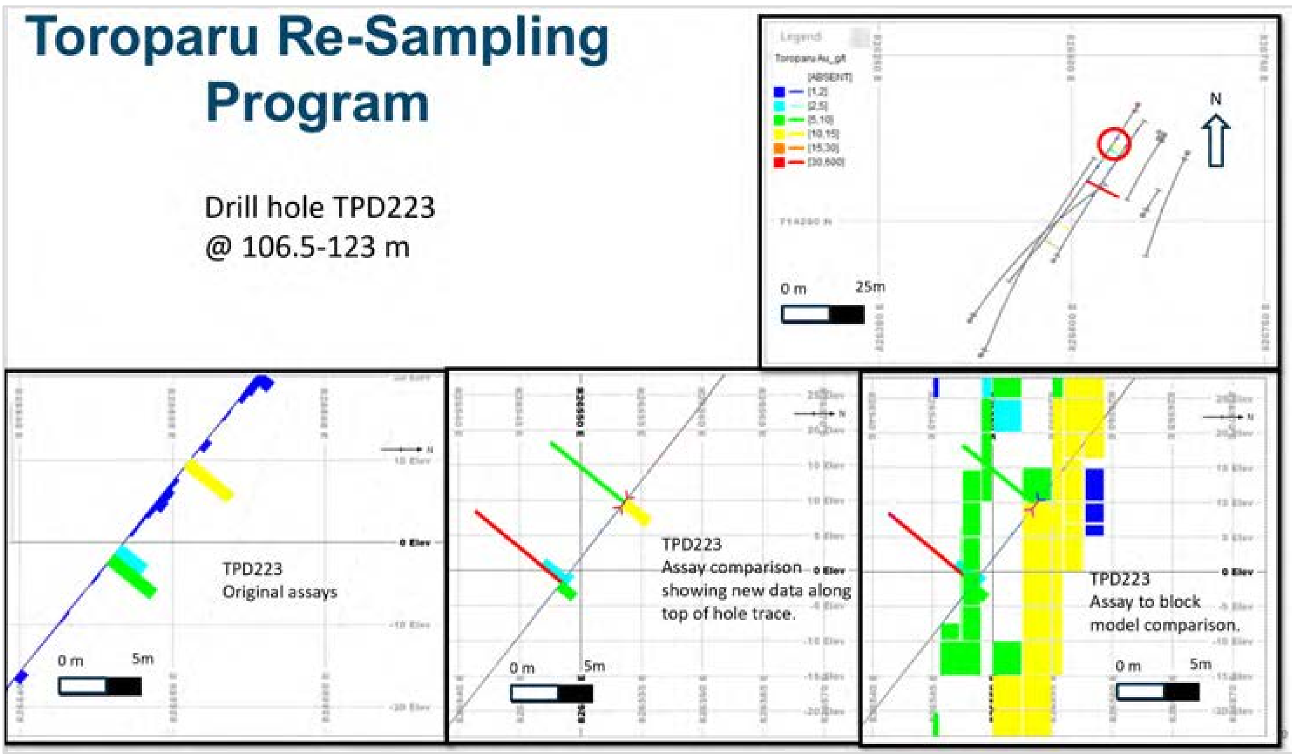


Figure 12-10: Re-sampling of drill hole TPD223 from 106.5 m to 123 m

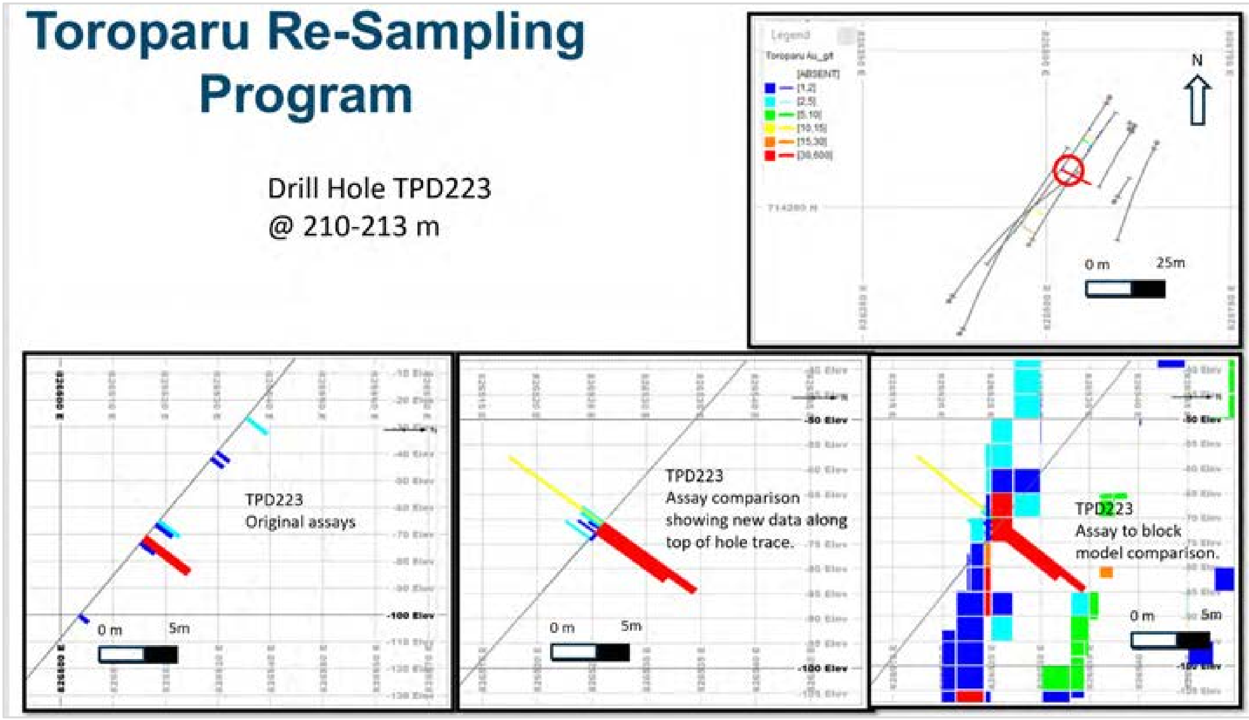


Figure 12-11: Re-sampling of drill hole TPD223 from 210 m to 213 m

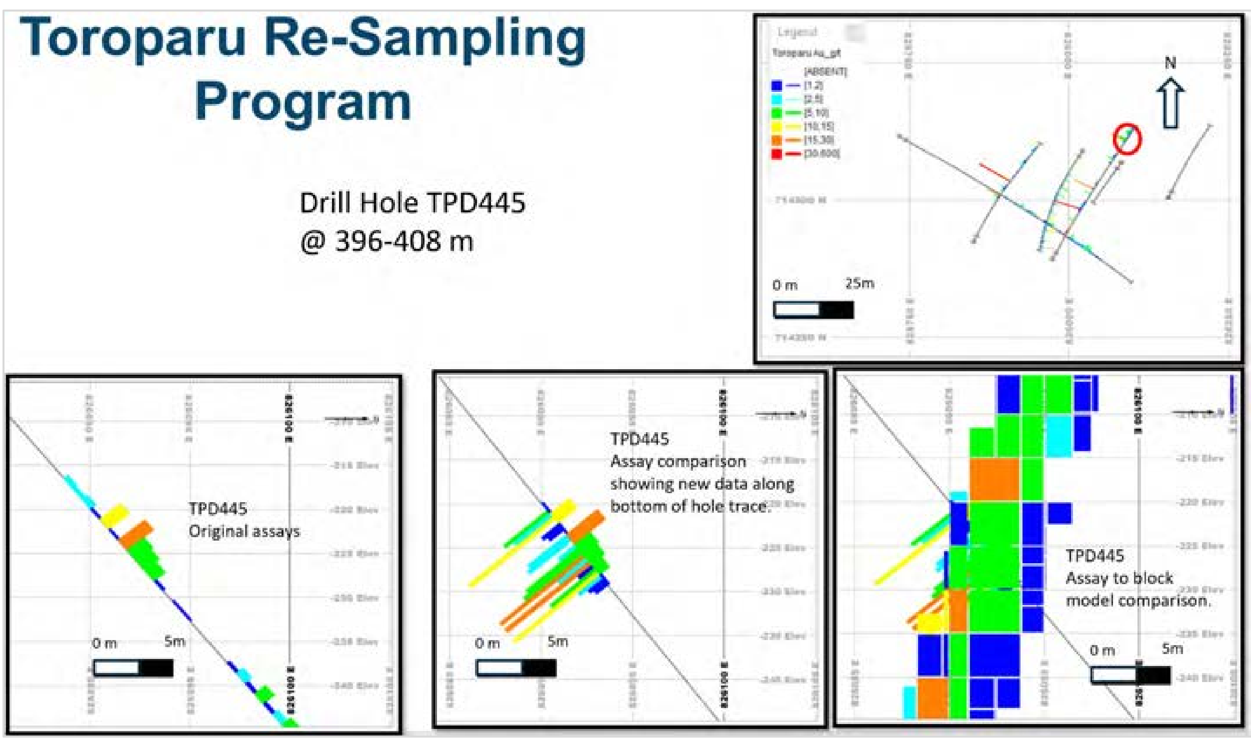


Figure 12-12: Re-sampling of drill hole TPD445 from 396 m to 408 m

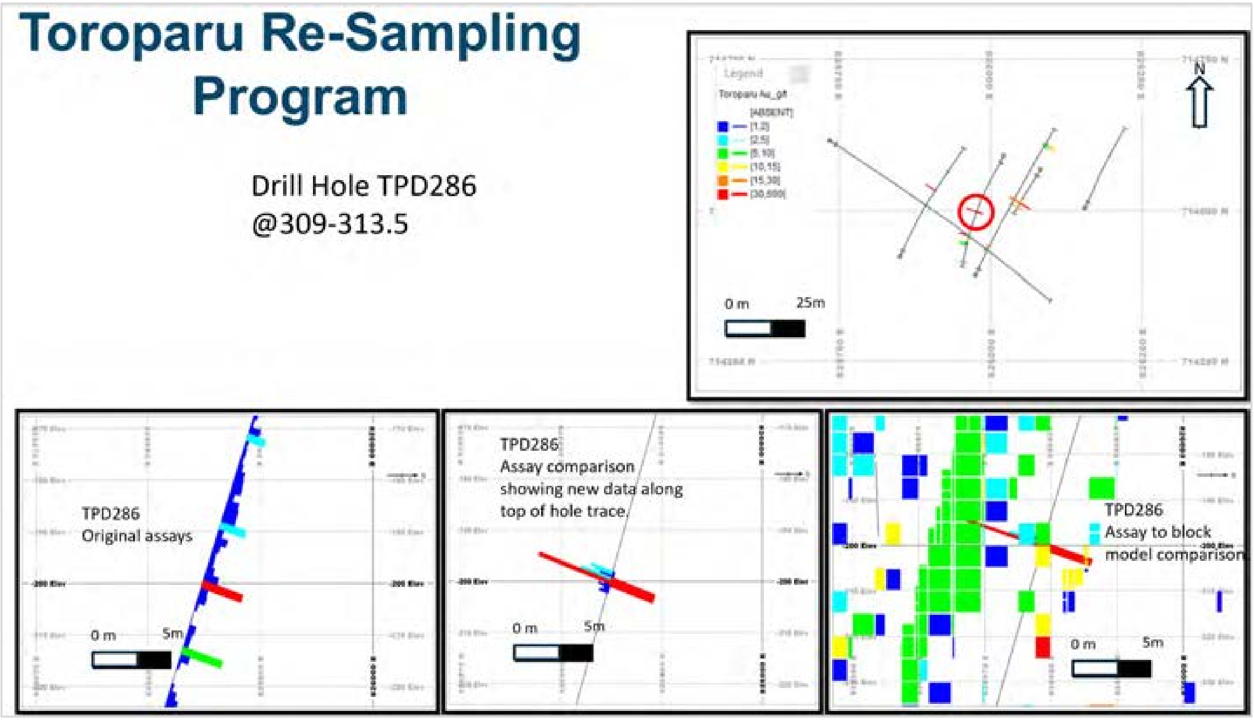


Figure 12-13: Re-sampling in drill hole TPD286 from 309 m to 313.5 m

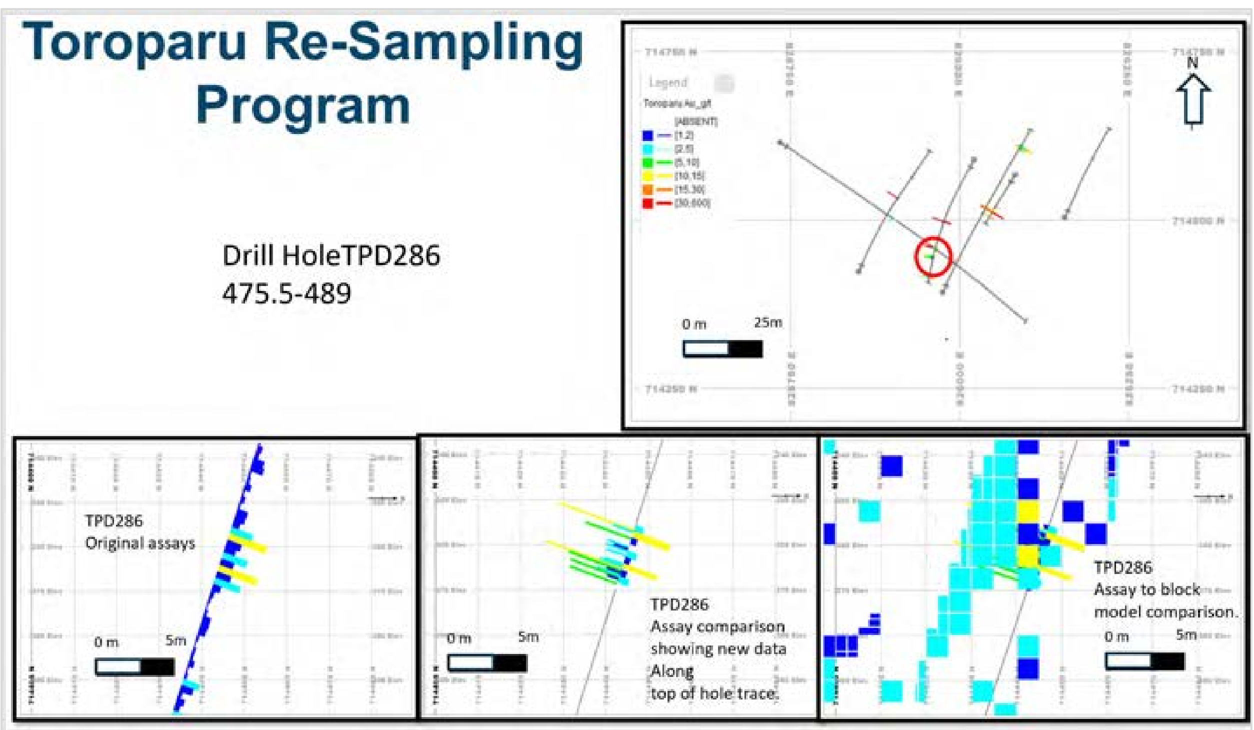


Figure 12-14: Re-sampling in drill hole TPD286 from 475.5 m to 489 m

12.1.4 Independent Sampling

The QP selected intervals from multiple Company drill holes for a total of 40 verification samples from the Toroparu Deposit Main and SE Areas (Table 12-1) and 34 verification samples from the Sona Hill Deposit (Table 12-2). The samples were marked with a logging pen, and the core was quarter cut to represent the same sample length and compared to pulps from previous assays. Both standards and duplicates were inserted within the verification sample order. During the visit to the core shack, Nordmin observed the cutting and sampling of the core selected by Nordmin to be quartered. The cutting and sampling were consistent with the Company's surface sampling procedures.

Table 12-1: Toroparu Deposit, Main, and SE Area Drill Holes Selected for Re-assaying

Hole ID	From (m)	To (m)	Previous Sample ID	Nordmin Check Sample ID
TPD535	222.74	223.22	978430	998597
TPD535	223.22	223.7	978432	998598
TPD535	223.7	224.2	978433	998599
TPD535	224.2	224.72	978434	998601
TPD535	224.72	225.21	978435	998603
TPD535	225.21	226	978436	998604
TPD535	226	226.5	978437	998605
TPD535	226.5	227	978438	998606
TPD535	227	227.5	978439	998607
TPD533	361	362	984844	998593
TPD533	362	362.5	984845	998594
TPD533	362.5	363.5	984847	998595
TPD190	26.5	28	643022	998590
TPD190	28	29.5	643023	998591
TPD164	52	53.5	645812	998585
TPD164	53.5	55	645813	998586
TPD164	55	56.5	645814	998587
TPD164	56.5	58	645815	998588
TPD553	227.5	228.5	993049	998577
TPD553	228.5	229.1	993050	998578
TPD553	229.1	229.62	993051	998579
TPD553	229.62	230.6	993052	998580
TPD553	230.6	231.6	993053	998581
TPD553	231.6	232.1	993054	998582
TPD553	232.1	233.12	993055	998584
TPD525	299	300.5	977868	998564

<table>
<tr><th>Hole ID</th><th colspan="2">From (m)</th><th colspan="2">To (m)</th><th>Previous Sample ID</th><th>Nordmin Check Sample ID</th></tr>
<tr><td>TPD525</td><td colspan="2">300.5</td><td colspan="2">301.31</td><td>977869</td><td>998565</td></tr>
<tr><td>TPD525</td><td colspan="2">301.31</td><td colspan="2">302</td><td>977870</td><td>998567</td></tr>
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<tr><td>41</td><td>41.5</td><td>998562</td></tr>
<tr><td>41.5</td><td>42</td><td>998563</td></tr>
</table>

Table 12-2 Sona Hill Deposit Drill Holes Selected for Re-assaying

Hole ID	From (m)	To (m)	Previous Sample ID	Nordmin Check Sample ID
SOD147	40.5	42	963465	998646
SOD147	39	40.5	963464	998645
SOD147	37.5	39	963463	998644
SOD147	36	37.5	963462	998642
SOD147	42	43.5	963467	998647
SOD130	57	57.82	962199	998632
SOD130	57.82	59.64	962200	998633
SOD130	59.64	60.37	962201	998634

Hole ID	From (m)	To (m)	Previous Sample ID	Nordmin Check Sample ID
SOD130	60.37	61.54	962203	998635
SOD130	61.54	63	962204	998636
SOD130	64.5	65.81	962206	998637
SOD130	65.81	67.04	962207	998640
SOD130	67.04	68.32	962208	998641
SOD114	192	193.06	954532	998625
SOD114	193.06	194.45	954533	998626
SOD114	194.45	195.45	954534	998627
SOD114	195.45	196.5	954535	998628
SOD114	288	289.5	954606	998629
SOD114	289.5	290.94	954607	998630
SOD114	290.94	292.5	954608	998630
SOD093	51.5	53	951605	998622
SOD093	53	53.7	951606	998623
SOD084	36.5	38	950954	998615
SOD084	38	39.5	950955	998616
SOD084	39.5	41	950956	998617
SOD084	41	42.5	950957	998618
SOD084	42.5	44	950959	998619
SOD084	44	45.5	950960	998620
SOD081	35.2	37	950731	998608
SOD081	37	38.47	950732	998609
SOD081	38.47	39.5	950733	998610
SOD081	39.5	41.2	950734	998612
SOD081	41.2	42.5	950735	998613
SOD081	42.5	43.88	950737	998614

Multiple intervals of samples were chosen to review assay results over larger areas and to test local variability. Since the intent of the proposed mining method is both open pit and underground, many intervals were checked with a goal to emulate benches or underground stopes that combine both the high-grade E-W domains and lower grade NW-SE domains. The chosen samples were over a range of low, medium, and higher-grade materials (Figure 12-15 and Figure 12-16).

Figure 12-15: DDH TPD325 between 35.5 m and 44.06 m outlining gold grades over an interval of approximately 8.56 m

Figure 12-16: DDH SOD147 33.44 m to 45.06 m

The QP assay results were compared to the Company database and were summarized in scatter plots for gold (Figure 12-17 and Figure 12-18). Despite some significant sample variance in a few samples, most assays were compared within reasonable tolerances for the mineralization types, and no material bias was evident (Table 12-3 and Table 12-4).

Table 12-3: Quarter Core Sampling Conducted by Nordmin on the Toroparu Deposit, Main, and SE Area Drill Holes, April 2021

Hole ID	From (m)	To (m)	Length	Half Core Au (ppm)	Quarter Core Au (ppm)
TPD535	222.74	223.22	0.48	0.006	0.007
TPD535	223.22	223.7	0.48	0.006	3.615
TPD535	223.7	224.2	0.5	7.312	21.300
TPD535	224.2	224.72	0.52	98.4	113.300
TPD535	224.72	225.21	0.49	9.461	2.797
TPD535	225.21	226	0.79	8.359	1.619
TPD535	226	226.5	0.5	2.15	0.297
TPD535	226.5	227	0.5	0.284	0.507
TPD535	227	227.5	0.5	0.337	0.601
TPD533	361	362	1	1.066	1.170
TPD533	362	362.5	0.5	33.9	39.600
TPD533	362.5	363.5	1	1.795	1.911
TPD190	26.5	28	1.5	1.569	1.062
TPD190	28	29.5	1.5	0.036	0.012
TPD164	52	53.5	1.5	1.222	2.351
TPD164	53.5	55	1.5	0.34	0.138
TPD164	55	56.5	1.5	0.101	0.068
TPD164	56.5	58	1.5	1.644	2.966
TPD553	227.5	228.5	1	0.243	0.175
TPD553	228.5	229.1	0.6	0.234	0.328
TPD553	229.1	229.62	0.52	15.7	17.800
TPD553	229.62	230.6	0.98	3.195	1.538
TPD553	230.6	231.6	1	0.667	0.616
TPD553	231.6	232.1	0.5	7.312	4.796
TPD553	232.1	233.12	1.02	0.287	0.385
TPD525	299	300.5	1.5	0.231	0.149
TPD525	300.5	301.31	0.81	0.231	0.248
TPD525	301.31	302	0.69	0.231	0.397
TPD525	302	303.22	1.22	0.231	0.390

<table>
<tr><th>Hole ID</th><th colspan="2">From (m)</th><th colspan="2">To (m)</th><th colspan="2">Length</th><th>Half Core Au (ppm)</th><th>Quarter Core Au (ppm)</th></tr>
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<tr><td rowspan="3">TPD325</td><td rowspan="3">37.5</td><td>37.5</td><td rowspan="3">39</td><td>38</td><td rowspan="3">1.5</td><td>0.5</td><td rowspan="3">0.495</td><td>0.092</td></tr>
<tr><td>38</td><td>38.5</td><td>0.5</td><td>0.108</td></tr>
<tr><td>38.5</td><td>39</td><td>0.5</td><td>1.075</td></tr>
<tr><td rowspan="3">TPD325</td><td rowspan="3">39</td><td>39</td><td rowspan="3">40.5</td><td>39.5</td><td rowspan="3">1.5</td><td>0.5</td><td rowspan="3">4.939</td><td>3.426</td></tr>
<tr><td>39.5</td><td>40</td><td>0.5</td><td>3.137</td></tr>
<tr><td>40</td><td>40.5</td><td>0.5</td><td>1.015</td></tr>
<tr><td rowspan="3">TPD325</td><td rowspan="3">40.5</td><td>40.5</td><td rowspan="3">42</td><td>41</td><td rowspan="3">1.5</td><td>0.5</td><td rowspan="3">1.211</td><td>2.778</td></tr>
<tr><td>41</td><td>41.5</td><td>0.5</td><td>1.439</td></tr>
<tr><td>41.5</td><td>42</td><td>0.5</td><td>0.768</td></tr>
</table>

Table 12-4: Quarter Core Sampling Conducted by Nordmin on the Sona Hill Deposit Diamond Drill Core, April 2021

Hole ID	From (m)	To (m)	Length (m)	Half Core Au (ppm)	Quarter Core Au (ppm)
SOD147	36	37.5	1.5	2.059	2.259
SOD147	37.5	39	1.5	1.383	1.555
SOD147	39	40.5	1.5	1.383	0.704
SOD147	40.5	42	1.5	0.702	2.223
SOD147	42	43.5	1.5	0.702	2.978
SOD130	57	57.82	0.82	0.306	0.145
SOD130	57.82	59.64	1.82	3.388	6.397
SOD130	59.64	60.37	0.73	0.04	0.880
SOD130	60.37	61.54	1.17	0.175	0.236
SOD130	61.54	63	1.46	0.036	0.055

Hole ID	From (m)	To (m)	Length (m)	Half Core Au (ppm)	Quarter Core Au (ppm)
SOD130	64.5	65.81	1.31	0.169	0.185
SOD130	65.81	67.04	1.23	2.48	3.420
SOD130	67.04	68.32	1.28	2.48	1.073
SOD114	192	193.06	1.06	0.027	0.034
SOD114	193.06	194.45	1.39	4.591	18.100
SOD114	194.45	195.45	1	0.469	0.512
SOD114	288	289.5	1.5	0.023	0.033
SOD114	289.5	290.94	1.44	0.237	0.136
SOD114	290.94	292.5	1.56	5.471	1.733
SOD093	53	53.7	0.7	1.632	1.547
SOD093	51.5	53	1.5	1.314	1.514
SOD084	36.5	38	1.5	2.768	2.656
SOD084	38	39.5	1.5	1.51	0.694
SOD084	39.5	41	1.5	0.013	0.032
SOD084	41	42.5	1.5	1.548	0.420
SOD084	42.5	44	1.5	0.133	0.092
SOD084	44	45.5	1.5	0.545	1.025
SOD081	35.2	37	1.8	0.561	1.350
SOD081	37	38.47	1.47	0.236	0.757
SOD081	38.47	39	0.53	4.48	2.101
SOD081	39	41.2	2.2	2.372	1.451
SOD081	41.2	42.5	1.3	0.782	0.754
SOD081	42.5	43.88	1.38	0.197	0.444

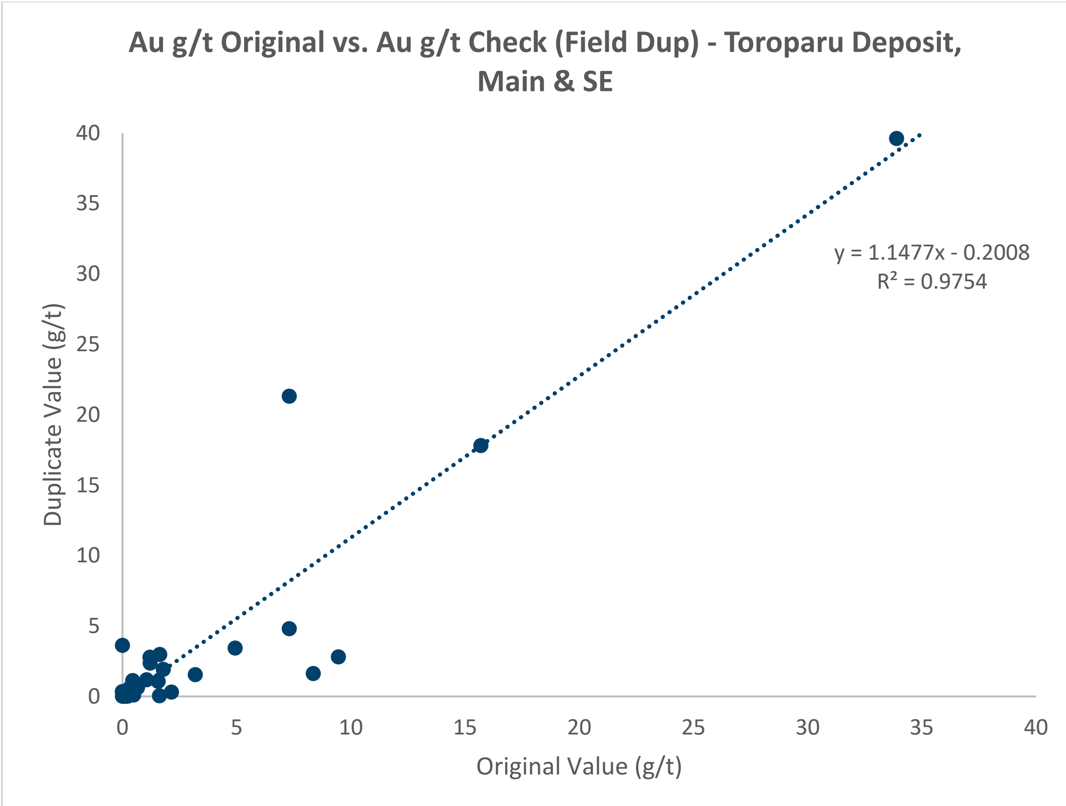


Figure 12-17: Scatter plot comparison of gold (g/t) verification drill core samples for the Toroparu Deposit, Main, and SE Areas One outlier not shown on chart is TPD535 224.2 m to 224.72 m (978434/998608) – see Table 12-1 and Table 12-2.

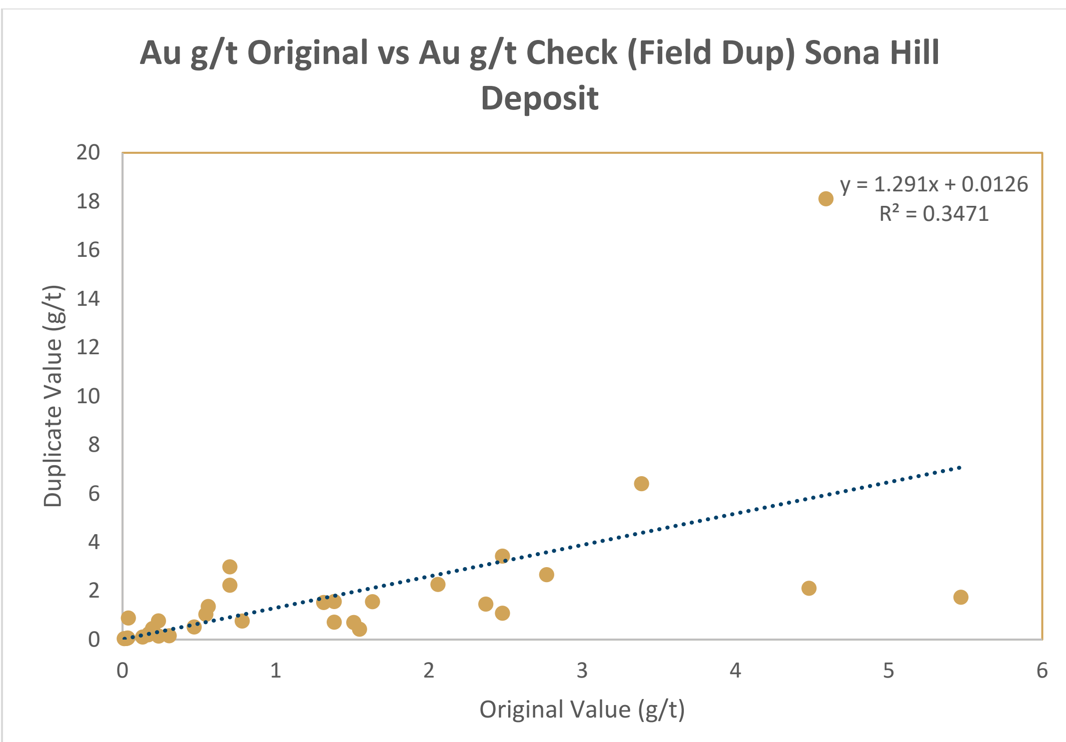


Figure 12-18: Scatter plot comparison of gold (g/t) verification drill core samples for Sona Hill Deposit

The drill core samples selected by the QP for verification analysis were individually placed into plastic sample bags, which were then packaged together and shipped to MS Analytical (Figure 12-19) for analysis using the Company's analytical procedures.

Figure 12-19: Verification samples sent to MS Analytical

The geologists use drill core as blank material that was drilled within waste rock within the Toroparu Deposit. The core has been assayed to confirm it has no gold within the interval (Figure 12-20).

Figure 12-20: Drill hole blank material from the Toroparu Deposit

12.1.5 Geological Interpretation, Surface Drilling, and Mineralized Surface Stockpiles Validation

The QP examined approximately 38 drill holes from the Toroparu Deposit, 16 holes from the Sona Hill Deposit and approximately 350 samples across the two deposits as part of the due diligence, including the recent 2021/2021 drilling campaigns (Figure 12-21, Figure 12-23 and Figure 12-24). In addition, various older and more recent 2020/2021 drill intervals were compared by the QP to the remaining core to determine if the logging and sampling intervals matched the information within the logs. The logging and sampling observed by the QP were consistent and representative.

Figure 12-21: Toroparu Deposit drilling locations of drill holes that the QP reviewed

Nordmin examined the rough thickness of the saprolite material in relation to the fresh rock material that is located within the proposed open pit and underground mining area. The geological logs confirm the expected depths of the saprolite and corresponding rock types (Figure 12-21, Figure 12-23 and Figure 12-24). Figure 12-22 illustrates the saprolite is approximately 25 m to 30 m in thickness within the Toroparu Deposit.

Figure 12-22: Hole TPD129, approximately 25 m to 32 m showing the saprolite and saprolite rock intervals

Various drill hole collar sites were reviewed, both historical collars disturbed by local artisanal miners and from the most recent 2020/2021 drill program. Figure 12-23 and Figure 12-24 illustrate the previous historical drill collars with the more recent infill 2020 drill program collar locations. The drill spacing from the field agrees with what is located within the drill hole database.

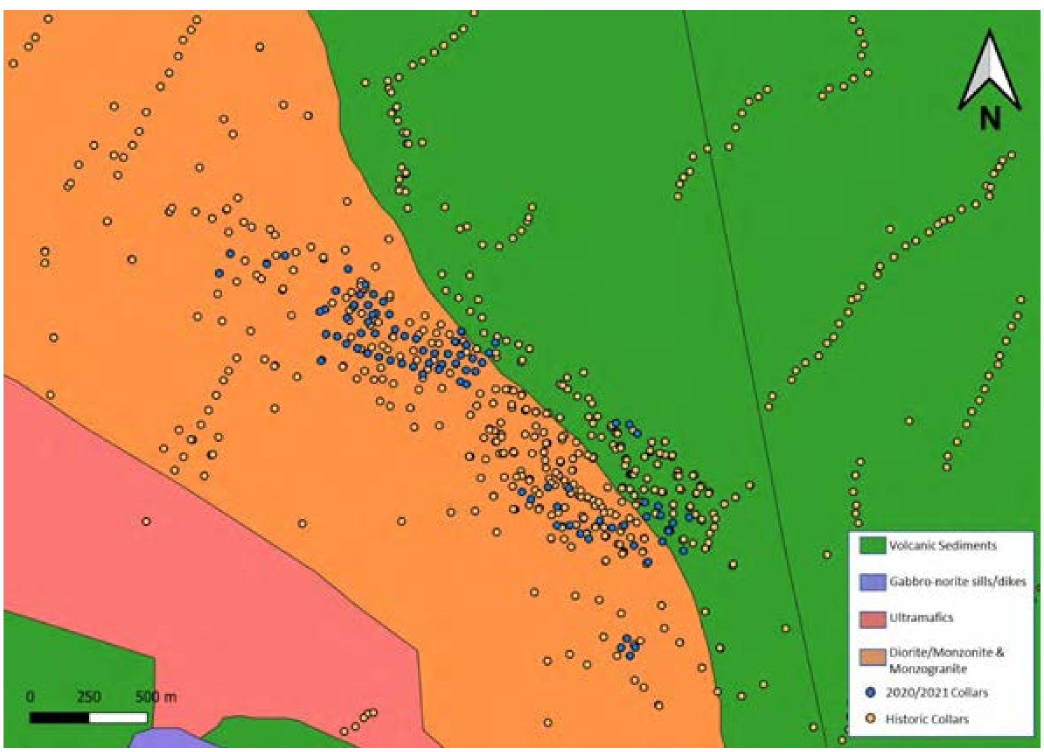


Figure 12-23: 2020/2021 drilling program collar locations versus historic drill hole collar locations within the Toroparu Deposit Main and NW Areas


Figure 12-24: Drill collar locations of TPD467 and TPD564 (2021)

In the spring of 2018, the Company became aware that several permanent drill hole collar markers had been disturbed or destroyed by surface alluvial placer mining that had occurred in the area of the Toroparu Deposit, Main, and SE Areas. The activity was conveyed as a legal alluvial mining operation of Alphonso but was conducted without the Company's knowledge and was confirmed after the damage had occurred. The alluvial disturbances resulted in shallow ponds that utilized floating gravity separation equipment. The shallow alluvial material was dredged and processed to varying depths estimated at 2 m to conceivably a maximum of 5 m. The result after the mining was a series of ponds and spoil piles covering the area, as shown in Figure 12-25.

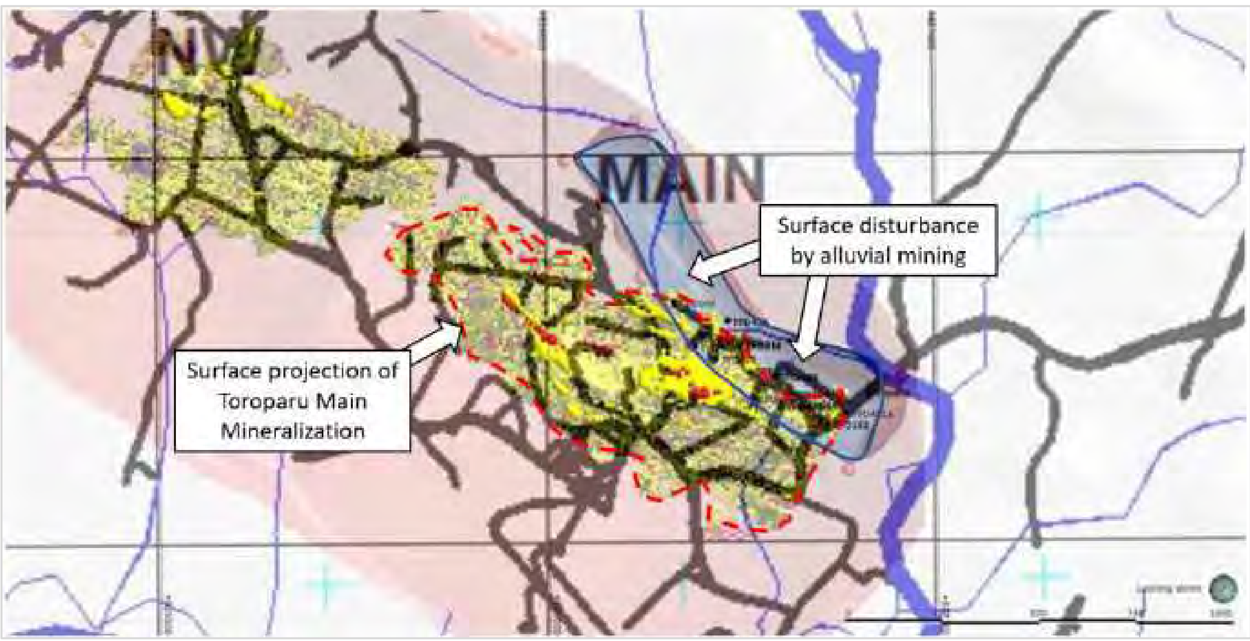
Figure 12-25: Aerial view of local open pits (mined out saprolite areas)

Nordmin reviewed the summary report of the alluvial disturbances from SRK's consulting QP opinion, and the Nordmin Consulting qualified person agrees with the following points:

- In total, 35 drill hole collar locations, identified in the field by pipe markers with drill hole collar ID's, have been determined by the Company to have been disturbed from their original position or are missing (Source: Sandspring, 2018; annotated by SRK
- Figure 12-26).
- Those 35 holes account for 4.5% of the total number of drill holes (788 in total), and the 9,593 m drilled in those 35 holes account for 4.6% of the total metres drilled in the Toroparu Deposit (including the Main, NW, and SE Areas).
- SRK conducted field examinations of the Project starting in 2011 and spot-checked several collar locations with handheld GPS units, as part of the data verification process. That work found that the collars checked, are as depicted on maps with appropriate coordinates, at the ± 2 m to 5 m accuracy of the handheld GPS units. At the time of the initial site visits, the drill hole collar markers that were missing were noted to be in place, and the on site review concluded that all drill hole collar markers were deemed to be in place.

SRK concluded the following:

- The missing collar markers were indeed once in place, as observed in the field by SRK QP's.
- The magnitude of the disturbance was relatively minor in terms of the number of holes and the total metreage affected.
- The disturbance may be of consequence to third party reviewers who might wish to verify all the surveyed collar locations for the Toroparu Deposit; in fact, others are still in place; and SRK considered the drill hole database to be valid and verified. Therefore, the missing collar markers were of little or no consequence with respect to the SRK Mineral Resource Estimate.
- Some locations were under water in residual ponds, so many of the collar locations would not be accessible. Any collar markers that were to be re-established were to be noted in the drill hole master database.
- For the SRK Mineral Resource Estimations and subsequent mine planning and Mineral Reserve Estimates, SRK considered the drill hole collars and the drill hole database to be valid and verified.



Source: Sandspring, 2018; annotated by SRK

Figure 12-26: Location of the surface disturbance by alluvial mining (conveyed in 2018)

12.2 Database Validation

Core sample records, lithologic logs, laboratory reports and associated drill hole information for all drill programs completed in the 2006 to 2021 period were digitally compiled for use in Geovia GEMS logger. Historical and current drilling program information was reviewed, and digital records of historical drilling were checked for both consistency and accuracy against original source documents available from the Company. The validation of approximately 20% of the assay dataset for sample interval and assay value information against corresponding source documents was carried out.

All drill hole data was compiled into a validated Microsoft Access® database that Nordmin reviewed digitally using a combination of Datamine and Target software programs. The QP completed a spot check verification of:

- Toroparu Deposit—920 (8%) of the lithologies, 740 (12%) of the geotechnical measurements, 8,358 (6%) of the assays.
- Sona Hill Deposit—146 (10%) of the lithologies, 109 (15%) of the geotechnical measurements, 7,306 (6%) of the assays.
- There were no errors found within the reviewed portion of the database.

The geology was validated for lithological units from the Company's Geovia GEMS logger. The geological contacts and lithology are aligned with the core contacts and lithology and are acceptable for use.

12.3 Review of the Company's QA/QC

The Company has a robust QA/QC process in place, as previously described in Section 11. The Company geologists actively monitor the assay results throughout the drill programs and summarize the QA/QC results, reporting weekly, and monthly. Most of the CRMs performed as expected within tolerances of two to three standard deviations of the mean grade. Therefore, Nordmin is satisfied that the QA/QC process performs as designed to ensure the assay data quality.

12.4 QP's Opinion

Upon completion of the data verification process, no apparent bias was determined between the historical programs compared to the programs since 2005. Therefore, it is the QP's opinion that the geological data collection and QA/QC procedures used by the Company are consistent with standard industry practices and that the geological database is of suitable quality to support the Mineral Resource Estimate.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

Multiple metallurgical test work programs starting in 2009 have yielded substantial information regarding the physical properties of the various feed grade mineralization in the Toroparu Deposit and their response to comminution, gravity concentration, rougher and cleaner flotation, and cyanide leaching.

In 2018, the Company started metallurgical test work on the Sona Hill Deposit. The Sona Hill Deposit comprises auriferous saprolitic material overlaying non oxidized hardrock which is also auriferous. The Toroparu Deposit similarly includes auriferous saprolite and hardrock zones as well as zones high in Cu.

For the Toroparu Deposit, test work has shown that both generalized mineralized material designations, Au mineralized material with ACO, also described elsewhere in the report as 'Au/Cu Ore' and Au mineralized material with LCO, also described elsewhere in the report as 'Au Ore', benefit from gravity concentration prior to further processing. Gravity Au recoveries of 38% were demonstrated for both ACO and LCO mineralized materials. (Note that the term "Ore" as used here is a naming convention dating back to the May 2013 PFS to identify two different categories of mineral processing materials and is not meant to convey positive economic connotations.)

Flotation recoveries achieved from ACO mineralized material were 91% Cu and 42% Au, in addition to gravity Au recoveries. Test work shows that both Cu and Au recoveries from LCO material were acceptable, but the relative loss in Au recovery versus a cyanide leach was not offset by an increased Cu flotation recovery.

Cyanide leach test work was conducted to determine the amenability of the ACO and LCO mineralized materials. It was determined that ACO flotation cleaner tailings and LCO gravity tailings leach recoveries were 8% and 58%, respectively, in addition to gravity and flotation recoveries.

Overall Au recoveries from ACO and LCO mineralized materials were determined to be 88% and 96%, respectively. These recoveries include gravity concentration, flotation, and cyanide leaching.

In addition to the primary hardrock ACO/LCO mineralized materials, saprolitic cover mineralized material was also tested for amenability to gravity concentration, flotation, and cyanide leaching. Gravity recovery test work indicate that >90% Au recoveries were achieved. Flotation recoveries for the saprolite cleaner test was 80%. Recoveries achieved for 72-hour whole mineralized material cyanide leaching was approximately 98% for both RoM saprolite fines and coarse saprolite ground to 80% passing (P_{80}) of 129 µm.

13.1 Metallurgical Testing

The historical testwork programs and reports are listed in Table 13-1.

Table 13-1: Historical Test Work Programs and Reports

Document or Test Program	Author or Laboratory	Date
An Investigation into the Recovery of Gold and Copper from the Toroparu Deposit of Sandspring Resources (Guyana) ("2009 SGS MetTest Program")	SGS Project No. 12039-001	June 22, 2009
A Preliminary Metallurgical Evaluation of the Master Composite from the Toroparu Deposit	SGS Project No. 12520-001 Nr. 1	September 9, 2011
A Metallurgical Evaluation of the Master Composite and Variability Samples from the Toroparu Deposit ("2011 SGS MetTest Program")	SGS Project No. 12520-001 Nr. 2	November 19, 2012
A Prefeasibility Study of Gold Ore Containing Low and Average Copper Grade from the Toroparu Deposit (2012 SGS MetTest Program)	SGS Project No. 12520-002	January 17, 2013
Prefeasibility Metallurgical Testing to Recover Gold on samples from Sandspring's Toroparu Project, Guyana	Inspectorate Project No. 1206809	December 2012
A Geotechnical Characterization of Tailings from Prefeasibility Testwork for the Toroparu Deposit	SGS Project No. 12520-002	February 25, 2013
Metallurgical Test Work Final Report on ACO and LCO. Composites from Sandspring Resources Toroparu Project in Guyana.	FLSmidth Dawson Metallurgical Laboratory Project No. P-14013 and P-14013AA	May 1, 2014
Metallurgical Assessment of the Toroparu Copper-Gold Project	ALS Kamloops Project No. KM4271	January 14, 2015
Further Metallurgical Assessment of the Toroparu Copper-Gold Project	ALS Kamloops Project No. KM4421	January 26, 2015
Metallurgical Testing of the Sona Deposit	Base Met Labs project/report no BL231	April 23, 2019
Additional Metallurgical Testing of the Sona Deposit	Base Met Labs project/report no BL473	December 11, 2019
Additional Metallurgical Testing of the Sona Deposit	Base Met Labs project/report no BL523	February 28, 2020

13.2 SGS 2009 Samples

13.2.1 SGS Lakefield Test Program 2009

The testwork conducted in 2009 by SGS Lakefield used a single composite sample and a single pulp sample. The single composite sample was made of nine hardrock samples which were crushed to pass 10 mesh and combined. The single pulp sample is taken from the saprolite gravity tailings mixture.

The head analyses of the two samples used for the metallurgical tests program in 2009 are provided in Table 13-2.

Table 13-2: Head Assays (2009)

Method	Elements (unit)	Hardrock Comp	Saprolite Tailing Mixture
Chemical Analysis	Ave. Au g/t	1.43	0.60
	Ag g/t	1.80	0.60
	Cu %	0.17	0.16
	S %	0.13	0.03
ICP Scan	Al g/t	81,000	100,000
	As g/t	<30	<30
	Ba g/t	190	260
	Be g/t	0.62	0.95
	Bi g/t	<20	<20
	Ca g/t	25,000	230
	Cd g/t	<2	<2
	Co g/t	16	22
	Cr g/t	<4	180
	Fe g/t	36,000	60,000
	K g/t	12,000	15,000
	Li g/t	<5	<20
	Mg g/t	8,400	5,700
	Mn g/t	300	290
	Mo g/t	<10	<5
	Na g/t	35,000	840
	Ni g/t	<20	42
	P g/t	530	320
	Pb g/t	<30	<20
	Sb g/t	<10	<10
	Se g/t	<30	<30
	Sn g/t	<40	<20
	Sr g/t	140	25
	Ti g/t	4,100	5,300
	Tl g/t	<30	<30
	U g/t	<20	<20
	V g/t	51	100
	Y g/t	14	12
	Zn g/t	59	29

Source: SGS, 2009

13.2.2 SGS Lakefield Test Program 2011

Five hardrock composite samples and three saprolite composite samples were prepared by SGS from drill core for the 2011 test program. The samples were selected to represent the various lithologies present in the mine and the overall production anticipated from the pit. Table 13-3 shows the sample identification along with the drill core mathematical grade averages for the interval selected.

Table 13-3: Head Samples Identification

Sample Description	Sample Number	Au, g/t	Cu %
Saprolite, Low-grade	1	0.182	0.083
Saprolite, Mid-grade	2	0.734	0.085
Saprolite, High-grade	3	0.898	0.072
Acid Intrusion, Average Grade	4	0.732	0.101
Massive Intermediate Volcanic, Low-grade	5	0.267	0.082
Massive Intermediate Volcanic, Mid-grade	6	0.565	0.127
Massive Intermediate Volcanic, High-grade	7	1.043	0.213
Massive Intermediate Volcanic, Master Composite	8	0.714	0.153

Source: SGS, 2011

The characteristic of the samples can be found in detail in SGS 2011 Met Test Program Sample Head Grade Assays (January 4, 2012, Inventory) however, a summary of the SGS analytical head grades is shown in Table 13-4.

Table 13-4: SGS Head Analyses

Sample Number	Au, g/t	Cu %
Sample #1	0.11	0.13
Sample #2	1.28	0.10
Sample #3	0.69	0.13
Sample #4	0.95	0.10
Sample #5	0.31	0.07
Sample #6	0.49	0.12
Sample #7	1.40	0.19
Sample #8	0.64	0.13

Source: SGS, 2011

The chemical analysis of the Master Composite, also known as Sample #8, showed the head grade to be 0.13% Cu, 0.17% S, 0.64 g/t Au and 1.7 g/t Ag. Additional elemental analysis for Sample #8 is shown in Table 13-5.

Table 13-5: Composite Sample #8 Analyses

Element	Value	Element	Value	Element	Value
Fe %	4.46	Cd g/t	<2	Sb g/t	<10
S %	0.17	Co g/t	20	Se g/t	<30
S= %	0.150	Cr g/t	65	Sn g/t	<20
Hg g/t	<0.3	K g/t	12,000	Sr g/t	175
Te g/t	<50	Li g/t	<20	Ti g/t	4,390
Ag g/t	<2	Mg g/t	13,000	Tl g/t	<30
Al g/t	73,600	Mn g/t	437	U g/t	<20
As g/t	<30	Mo g/t	<10	V g/t	84
Ba g/t	219	Na g/t	29,800	Y g/t	14.6
Be g/t	0.76	Ni g/t	33	Zn g/t	55
Bi g/t	<20	P g/t	688	Sb g/t	<10
Ca g/t	33,400	Pb g/t	<20	Se g/t	<30

Source: SGS, 2011

13.2.3 SGS Lakefield Test Program 2012/2013

The metallurgical test program conducted by SGS Lakefield in 2012 focused on investigating two main mineralized material types from the Toroparu Deposit. These mineralized material types were classified as Au mineralized material with ACO and Au mineralized material with LCO.

ACO

An ACO Composite sample was prepared from six samples, five of which were remaining from a previous test program. The weights and metal grades of the individual components are given in Table 13-6.

Table 13-6: ACO Sample Composition

Sample Number	Wt, kg	Grade	
		Au, g/t	Cu %
New Sample	120	2.00	0.29
Sample # 4	23.2	0.95	0.10
Sample # 5	13.7	0.31	0.07
Sample # 6	41.6	0.49	0.12
Sample # 7	27.8	1.40	0.19
Sample # 8	118.3	0.64	0.13
Total	**344.6**	**1.16**	**0.18**

Source: SGS, 2012

The ACO Composite was submitted for Au head grade determination using the screened metallics protocol. The calculated Au head grade was 1.18 g/t.

The ACO Composite was also submitted for Ag, S, Cu, and cyanide soluble Cu analysis in triplicate. The results of the analysis are shown in Table 13-7. The average grades for the ACO Composite were 0.19% for Cu, 2.5 g/t for Ag and 0.21% for S. The cyanide soluble Cu average was 0.037%, which is approximately 20% of the total Cu present in the ACO Composite. The results of the ICP scans are displayed in Table 13-8.

Table 13-7: Assay Results

Sample	Cu %	Ag g/t	S %	Cu NaCN %
ACO Composite	0.19	2.2	0.21	0.035
	0.18	2.3	0.22	0.037
	0.19	3.1	0.21	0.039
Average	**0.19**	**2.5**	**0.21**	**0.037**

Source: SGS, 2012

Table 13-8: ICP Scans

Assay	ACO Composite		
Al g/t	70,400	70,200	69,400
As g/t	<30	<30	<30
Ba g/t	250	249	247
Be g/t	0.64	0.64	0.64
Bi g/t	<20	<20	<20
Ca g/t	30,900	31,000	30,600
Cd g/t	<2	<2	<2
Co g/t	21	21	21
Cr g/t	91	89	79
Fe g/t	56,400	39,900	39,400
K g/t	12,900	13,100	13,800
Li g/t	13	13	13
Mg g/t	11,400	11,200	11,300
Mn g/t	406	403	392
Mo g/t	5	13	7
Na g/t	25,900	24,200	24,800
Ni g/t	25	26	24
P g/t	632	607	586
Pb g/t	<20	<20	<20
Sb g/t	<10	<10	<10
Se g/t	<30	<30	<30
Sn g/t	<20	<20	<20
Sr g/t	158	157	158
Ti g/t	3,910	3,900	3,890

Assay	ACO Composite		
Tl g/t	<30	<30	<30
U g/t	<20	<20	<20
V g/t	73	71	69
Y g/t	16.3	14.0	14.3
Zn g/t	63	60	58

Source: SGS, 2012

LCO

Five samples were provided for the LCO testwork program. The samples were labelled as follows:

- Main Deposit NW End (LCO Variability Ore A);
- Main Deposit SE End (LCO Variability Ore B);
- Main Deposit S End;
- Startup Grade (LCO Variability Ore C); and
- South East Satellite Deposit (LCO Variability Ore D).

The samples were stage-crushed to 100% minus 1/2 inch. An LCO Master Composite of 150 kg was prepared using the Main Deposit samples, by combining 77.2 kg of NW End, 66.9 kg SE End and 5.8 kg S End. Four of the five samples were also used as Variability Samples. The Master Composite and Variability samples were stage-crushed to -10 mesh and riffled into 2 kg and 10 kg test charges. The remaining material was combined as a High Pressure Grinding Roll (HPGR) Composite and retained at minus 1/2 inch and composed of 199.3 kg NW End, 168.1 kg SE End and 17.7 kg S End.

The head grades of the sample are shown in Table 13-9.

Table 13-9: Head Grades

Sample	Au g/t	Ag g/t	Cu %	S %
LCO Master Composite	0.74	0.8	0.071	0.12
LCO-A Variability Composite (NW End)	0.55	<0.5	0.056	0.09
LCO-B Variability Composite (SE End)	0.62	1.0	0.084	0.15
LCO-C Variability Composite (Startup)	1.52 *	0.5	0.050	0.09
LCO-D Variability Composite (SE Deposit)	1.12	<0.5	0.037	0.10

Source: SGS, 2012

* Direct Head assay was 0.88 g/t, revised as average of Calculated Head of Tests G-9 and G-17

13.2.4 Inspectorate Test Program 2012

The metallurgical test program conducted by Inspectorate Exploration and Mining Services (Inspectorate) in 2012 focused on investigation of Au recovery from saprolitic mineralized material from the Toroparu Deposit.

A total of 210 individual sample bags with approximate wet weight of ~600 kg was received from ACME Laboratories on September 13, 2012.

Samples were sorted into a saprolite zone composite and a transition zone composite. Individual samples received and identified as core in the Sample Receiving Log were stored and were not included in this part of the test program.

A list of 90 individual sample bags selected as per the compositing details provided by SRK to Inspectorate for the saprolite zone composite preparation is presented in Table 13-10.

Table 13-10: Inspectorate Saprolite Composite Test Sample Labels

Pail ID	Sample ID	Pail ID	Sample ID	Pail ID	Sample ID	Pail ID	Sample ID	Pail ID	Sample ID
TM-001	687975	TM-001	694286	TM-004	511244	TM-008	622345	TM-009	680376
TM-001	687976	TM-001	694287	TM-004	511245	TM-008	622346	TM-009	380377
TM-001	687977	TM-001	694288	TM-004	511246	TM-008	622347	TM-009	680378
TM-001	687978	TM-001	694289	TM-004	511247	TM-008	622349	TM-009	680379
TM-001	687980	TM-001	688687	TM-004	511248	TM-008	622350	TM-009	680381
TM-001	687981	TM-001	688689	TM-004	511249	TM-008	622351	TM-009	680382
TM-001	684708	TM-001	688690	TM-004	511250	TM-008	622352	TM-009	680383
TM-001	684710	TM-001	688691	TM-004	511251	TM-008	622353	TM-009	680384
TM-001	684711	TM-001	688692	TM-004	511252	TM-008	622354	TM-009	680385
TM-001	684712	TM-001	688694	TM-004	511253	TM-008	622355	TM-009	680386
TM-001	684713	TM-001	688695	TM-004	511254			TM-009	680387
TM-001	684715	TM-001	688696	TM-004	511255			TM-009	680389
TM-001	684716	TM-001	688697	TM-004	511256			TM-009	680390
TM-001	684717	TM-001	688698	TM-004	511257			TM-009	680391
TM-001	684718	TM-001	688699	TM-004	511258			TM-009	680392
TM-001	684719	TM-001	688700	TM-004	511259			TM-009	680393
TM-001	684720	TM-001	688702					TM-009	680394
TM-001	694276	TM-001	688703						
TM-001	694278	TM-001	688704						
TM-001	694279	TM-001	688705						
TM-001	694280	TM-001	688706						
TM-001	694281	TM-001	688707						
TM-001	694283								
TM-001	694284								
TM-001	694285								

Source: Inspectorate, 2012

Each composite was low temperature dried, bigger lumps were broken by rolling without crushing and then thoroughly blended by riffling three times before splitting into test charges and head assay aliquots. Triplicate splits from each sample were pulverized and assayed for Au, Ag, S speciation, C speciation, Hg and ICP. Another triplicate split of ~500 g from each sample was subjected to metallic screen. A total of 191 kg of saprolite composite was prepared, and a total of 68 kg of transition composite was prepared.

Two separate saprolite composites were created in the program, identified as saprolite and transition. In addition, the saprolite composite was split into coarse and fine fractions. The transition composite did not undergo any metallurgical testing other than head grade analysis.

The head grades of the sample are shown in Table 13-11.

Table 13-11: Head Grades

Sample ID	Average Triplicate Pulverized Splits(g/t)		Average Triplicate Metallics (g/t)		Scrubbing Screen Analysis (g/t)		Average Measured Head (g/t)	
	Au	Ag	Au	Ag	Au	Ag	Au	Ag
Saprolite Composite	0.83	2.40	1.24	2.61	1.32	4.50	1.13	3.17
Transition Composite	0.73	2.11	1.62	2.75	0.70	4.25	1.02	3.04
Saprolite Coarse	3.30	6.92	-	-	-	-	-	-
Saprolite Fines	0.71	2.40	-	-	-	-	-	-

Source: Inspectorate, 2012

13.2.5 Compositing Strategy 2014

General

Test work was conducted at FLSmidth Dawson Laboratories (Salt Lake City) and the ALS Laboratory in Canada (Kamloops) during 2014. The work was based on a series of metallurgical samples selected by the Company from the exploration and resource definition drilling. The samples were predominantly

½ NQ core as well as a lesser number of ½ HQ core samples.

Five samples from SGS Canada as had been used in the 2012 test work program at the SGS facility were also sent to FLSmidth Dawson Metallurgical for bulk mineralogy and comminution test work. These composites referred to as Sample 4, 5, 6, 7, and 8. Sample 8 being a Master Composite.

Fresh Material (Non Oxidized)

The nominally 640 ½ NQ core samples were sent to FLSmidth Dawson Metallurgical Laboratory who were responsible for preparing composites as instructed by the Company. Various lithological and spatial (location) composites were prepared as well as composites to explore variability. A selection of samples and composites were sent from the FLSmidth Dawson Metallurgical Laboratory facility to ALS Laboratories to allow ALS to undertake work on the same sample set.

From these core samples there were also a set of intervals set aside for high pressure grinding rolls testing. As this work was not conducted, these intervals were included for use to make up spatial composites (SC) as described below.

The ½ NQ core had generally been assayed in 1.5 m intervals. The core samples sent to FLSmidth Dawson Metallurgical Laboratory had been packed so that 3 m of core (two lots of 1.5 m intervals) were packed together and mixed. In some instances, three lots of 1.5 m intervals had been packed as one sample. The as packed sample grades were estimated by averaging the assays of the intervals combined per sample. This assumed similar mass for each 1.5 m interval mixed together. The samples were to be mixed/composited and subjected to head assay later and therefore this averaging was considered appropriate given the calculated assay was only to be used for composite recipe estimates.

The lower grade samples of less than 0.001 g/t Au and 0.005% Cu were identified and were not utilized in compositing.

The cut-off between the LCO and ACO materials had been defined previously to be 0.09% Cu. Further consideration of this cut-off was made, and it was considered that it was not necessarily the most appropriate cut-off grade due to dependency of the revenue of each block of mineralized material being a function of the Au grade, Cu grade, the respective recoveries, and the operating cost to process. In the case of the higher Cu grade mineralized materials, this was influenced by the actual amount of Cu that was cyanide soluble.

It had been shown that a lower grade Cu mineralized material with high cyanide soluble levels of Cu may be more suited to flotation than a higher-grade Cu mineralized material (with the same Au grade) that did not have a high cyanide solubility of Cu. The precise cut-off grade based on a Cu assay alone did not present the full picture. Consequently, until the Cu-cyanide-Au relationship could be defined by more specific test work, it was decided that the first pass ACO and LCO composites would have a wider band of Cu grade assumed. LCO Composite recipes were therefore produced at less than 0.05% Cu and ACO at +0.15% Cu for the initial round of testing.

As the dominant lithologies were volcanic (V) and intrusive (IN), composite recipes considered these lithologies along with the ACO and LCO categorization. In addition, the years in which the mineralized material blocks represented by the various samples were expected to be processed based on the mine schedule of the time were defined. The samples available being split into two “year” categories.

This resulted in a 2 x 2 x 2 matrix of lithological composites to give a total of eight composites. (ACO/LCO) x (lithology V or IN) x (early or late in production schedule). The early or late production samples in part combined to provide an ACO-V and ACO-IN composite as well as an LCO-V and an LCO-IN composite.

These composites were prepared at a coarse crush to allow comminution testing to be conducted. They were then further reduced in size for bond ball work index testing and leaching or flotation test work as well as mineralogical study.

The ½ HQ core samples were similarly dispatched to FLSmidth Dawson Metallurgical Laboratory for compositing and used for crushing work index (CWi) test work.

FLS was requested to prepare a series of fresh mineralized material (non oxidized) sub composites referred to as SC. The various intervals used to put these SC together being located grouped along strike of the proposed open pit at various ranges of depth. In addition, these SC were selected with consideration of Au grade, Cu grade, period in the proposed production schedule or lithology. Not all of these parameters were maintained by all SC. Location and grade were maintained but at times the compositions were varied to reflect the variable of time in production or lithology, but not necessarily both. Noting that some of the SC included the other minor lithologies present, dacite and dyke material.

In all, 41 SC were prepared. The composites are summarized by Figure 13-1.

Sub-comp	Yr	Lith	Wt, kg	Au, g/t	Cu, %	Section From	To	Depth From	To
SC 1	7	V	17.55	0.982	0.049	450 W	700 W	50	143
SC 2	16	V	18.00	0.692	0.039	450 W	700 W	189	312
SC 3	16	IN	26.76	0.788	0.043	450 W	700 W	360	562
SC 4	7		24.00	1.005	0.055	250 W	400 W	55	161
SC 5	7		21.00	1.557	0.039	250 W	400 W	165	293
SC 6			30.92	0.934	0.045	250 W	400 W	305	408
SC 7	16	IN	13.36	0.981	0.040	250 W	400 W	443	517
SC 8	7	IN	19.00	1.866	0.051	50W	200 W	68	140
SC 9	7		30.00	0.977	0.056	50W	200 W	168	282
SC 10	16		11.00	0.415	0.062	50W	200 W	168	282
SC 11	16		30.92	0.660	0.047	50W	200 W	305	423
SC 12	16		15.44	0.626	0.064	50W	200 W	305	423
SC 13			20.00	0.827	0.060	150 E	0	47	147
SC 14			27.75	1.020	0.045	150 E	0	154	302
SC 15			16.13	1.144	0.054	150 E	0	154	302
SC 16			24.00	0.824	0.049	150 E	0	312	368
SC 17			18.93	1.673	0.040	300 E	200E	54	180
SC 18	7		25.39	1.889	0.036	1125 E	925 E	35	101
SC 19	16		34.72	1.565	0.021	1125 E	925 E	102	206
SC 20	5, 7		9.80	1.488	0.241	450 W	800 W	54	95
SC 21	16		12.00	1.264	0.187	450 W	800 W	267	362
SC 22	16		12.00	2.219	0.340	450 W	800 W	363	527
SC 23	5		20.00	1.117	0.225	250 W	400 W	39	148
SC 24	5		19.99	1.231	0.354	250 W	400 W	166	253
SC 25			39.22	0.831	0.121	250 W	400 W	255	389
SC 26			21.00	1.058	0.243	250 W	400 W	255	389
SC 27			15.00	3.018	0.451	250 W	400 W	255	389
SC 28	16	IN	20.30	1.768	0.164	250 W	400 W	409	532
SC 29			12.00	0.881	0.177	50 W	200 W	63	147
SC 30	5		11.23	1.370	0.472	50 W	200 W	63	147
SC 31	5		15.50	1.620	0.174	50 W	200 W	164	269
SC 32	5		10.00	2.478	0.490	50 W	200 W	164	269
SC 33	16		14.97	1.71	0.12	50 W	200 W	277	398
SC 34	16		15.00	1.87	0.31	50 W	200 W	277	398
SC 35			17.60	1.46	0.20	200 E	0	33	180
SC 36			23.66	1.05	0.15	200 E	0	240	381
SC 37			12.00	0.88	0.09	1075 E	925 E	35	80
SC 38			25.90	1.09	0.09	500 W	700W	Various	
SC 39			34.16	1.06	0.17	300 W	400 W	Various	
SC 40			28.95	1.38	0.20	50 W	200 W	Various	
SC 41			22.18	1.36	0.14	200 E	0	Various	
HPGR Samples									

Source: FLS, 2014

Figure 13-1: Spatial composite summary

The SC provided the building blocks for the various Au leaching and flotation composites to be generated, as well as providing samples for variability testing in their own right. That is, the intent being to use the SC for variability work once the general flowsheet is defined.

From the SC, a series of Fresh Composites were generated. These consisted of:

- Four Au SC for use in Au leaching work (lower Cu grades);
- Four Copper SC for use in flotation work (higher Cu grades);
- A bulk Au composite for use in those tests that required larger sample volumes such as carbon kinetics and settling;
- A "GRG" or Gravity Recoverable Au composite which was used for gravity Au assessments;
- A "Cu Bulk" composite for use in those tests that required larger sample volumes such as settling work as well as filtration of concentrates;
- Two "Cu Lock Cycle" composites for Locked Cycle flotation testing; and
- Three "Grade Variability" composites to explore the impact of grade on metallurgical responses to the flowsheet options investigated.

From the remaining SC, the strategy was to use the discrete remnants and subject them to the final Au leaching and flotation flowsheets to establish behaviour associated with low and high-grade Au and Cu combinations. Both low and high Cu were to be subjected to leaching and flotation. The outcomes were expected to assist in the designation of the criteria which would eventually direct one mineralized material block or another to the most appropriate flowsheet route.

Unfortunately, the laboratory mixed many of the discrete SC together believing the flowsheet Variability samples were supposed to be combined into a "Leach Variability" and a "Flotation Variability" composite. As a consequence of this, there were fewer SC available for variability assessments at the end of the program. The Leach Variability and Flotation Variability Composites were used for other work where larger sample sizes were advantageous.

Saprolitic and Transitional Samples

A selection of saprolite and transitional core samples were sent to ALS Kamloops via FLSmidth Dawson Metallurgical Laboratory. Two saprolitic SC and four transitional SC were generated based on recipes provided by the Company.

In addition, a second set of saprolite and transitional core samples were dispatched to ALS Kamloops direct. Saprolitic SC and transitional mineralized material SC were generated again based on recipes provided by the Company.

13.2.6 FLS Test Program 2014

A set of ½ HQ core samples were dispatched for CWi tests.

Five samples from SGS Canada identified as Samples 4, 5, 6, 7, and 8 from 2012 test work were prepared for bulk mineralogy and comminution work.

Eight metallurgical sub composites representing ACO-V (Yr 0-5 and 5-16), ACO-IN (Yr 0-5 and 5-16), LCO-V (Yr 0-7 and 7-16) and LCO-IN (Yr 0-7 and 7-16) were constructed according to the Company's instructions. Each sub composite was further split out to make four metallurgical test composites, identified as ACO-V, ACO-I, LCO-V and LCO-I.

Select samples were subjected to GRG testing as well as settling work.

The balance of the samples was prepared and composited according to the Company's instructions and dispatched to ALS Laboratories for further metallurgical test work.

13.2.7 ALS Program 2014

ALS conducted two test work programs. Initially, program KM4271 was undertaken which was focused on the fresh mineralized material samples. Program KM4421 was undertaken and focused on the saprolitic and transitional samples.

The ALS work included:

- Fresh samples: Chemical content assessment, mineralogy, mineral fragmentation, flotation, cyanidation, diagnostic leaching, activated carbon performance criteria determination, flocculant screening, settling and viscosity measurements; and
- Saprolite and Transitional samples: Chemical content assessment, gravity and cyanidation including cyanide soluble Cu assessment for the transitional samples.

13.3 Mineralogy

13.3.1 Sulphide and Gangue Minerals

In the 2011 metallurgical test program, SGS conducted mineralogical studies on the Master Composite (Sample #8).

The investigation used QEMSCAN® Particle Mineral Analysis as a means of determining the mineralogical content and fragmentation properties of the composite sample. The mineralogical breakdown is shown in Table 13-12.

Table 13-12: Mineral Composition and Copper Elemental Deportment of Master Composite

Mineral Composition (%)				Copper Deportment (%)	
Sulphide Minerals	**Mass**	**Gangue Minerals**	**Mass**	**Copper Bearing Minerals**	**Mass**
Chalcopyrite	0.48	Quartz	28.79	Chalcopyrite	83.0
Covellite	0.01	Plagioclase	20.90	Covellite	2.51
Bornite	0.04	K-Feldspar	2.41	Bornite	13.0
Pyrite	0.13	Amphiboles	1.21	Other Sulphides	1.45
Other Sulphides	0.02	Muscovites/Sericite	10.31		
		Biotite	0.64		
		Chlorite	12.68		
		Clay	12.05		
		Epidote	2.26		
		Sphene	1.00		
		Other Silicates	0.34		
		Fe-Oxides	0.18		
		Other Oxides	0.12		
		Calcite	5.68		
		Other Carbonates	0.11		
		Apatite	0.52		
		Fluorite	0.02		
		Sulphates	0.00		
		Other	0.09		
Total	**0.69**	-	**99.31**	-	**100**

Source: SGS, 2012

The primary information collected from the QEMSCAN examination included both modal, and liberation and association data. The QEMSCAN study involved stage-pulverizing a subsample of Sample #8 to a size of -250 µm. The sample was then separated into size fractions of +212 µm, - 212/+150 µm, -150/+75 µm, -75/+20 µm and -20 µm.

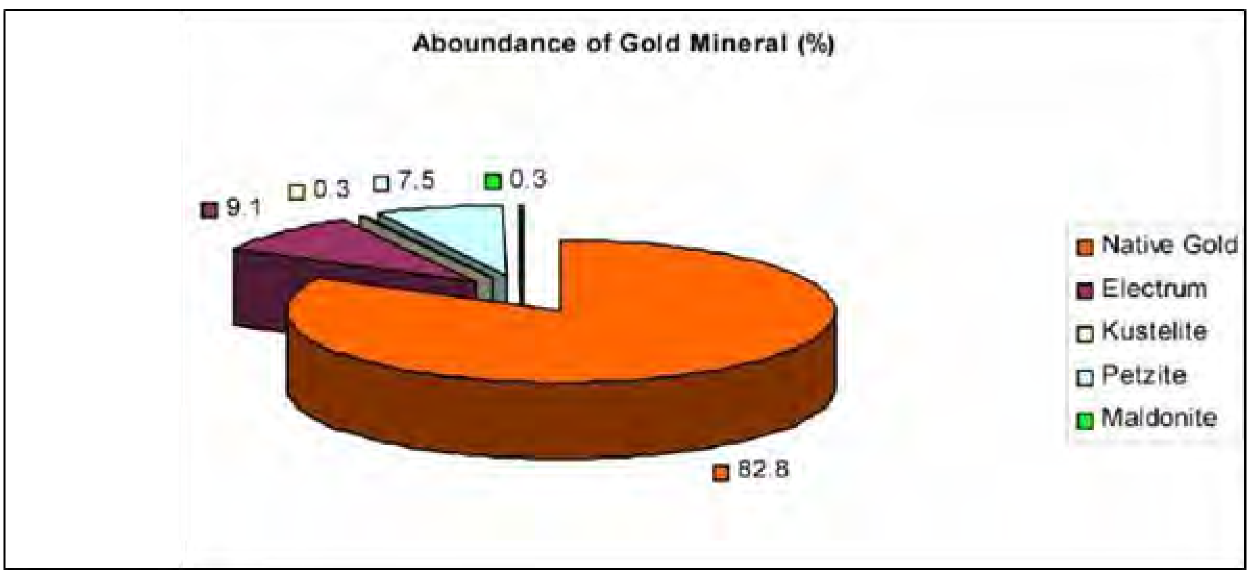
The modal analysis shows that the sample is dominated by silicates with chalcopyrite (0.48%) being the sulphide mineral with the greatest content. Other minerals of interest in the Cu minerals, covellite (0.01%) and bornite (0.04%), and pyrite (0.13%) were also identified but present in smaller quantities. The major silicate minerals are quartz (28.8%), plagioclase (20.9%), chlorite (12.7%), clays (12.1%),muscovite/sericite (10.3%) and calcite (5.9%).

Liberation and association for the Cu minerals and pyrite show that the liberation of the Cu minerals improves substantially at grinds finer than 150 µm with 83.3%, 92.0% and 95.0% of the Cu minerals free and liberated in the -150/+75 µm, -75/+20 µm and -20 µm size fractions respectively, while the Cu minerals are 0.5% and 22.5% free and liberated in the +212 µm and -212/+150 µm size fractions respectively. In contrast, the pyrite mineralization is widely distributed with significant liberation at all size fractions in the range of 80% and over 90% in some cases. Interestingly, the Cu minerals and pyrite have negligible mutual association so that producing a marketable concentrate was expected to be possible.

13.3.2 Gold Deportment

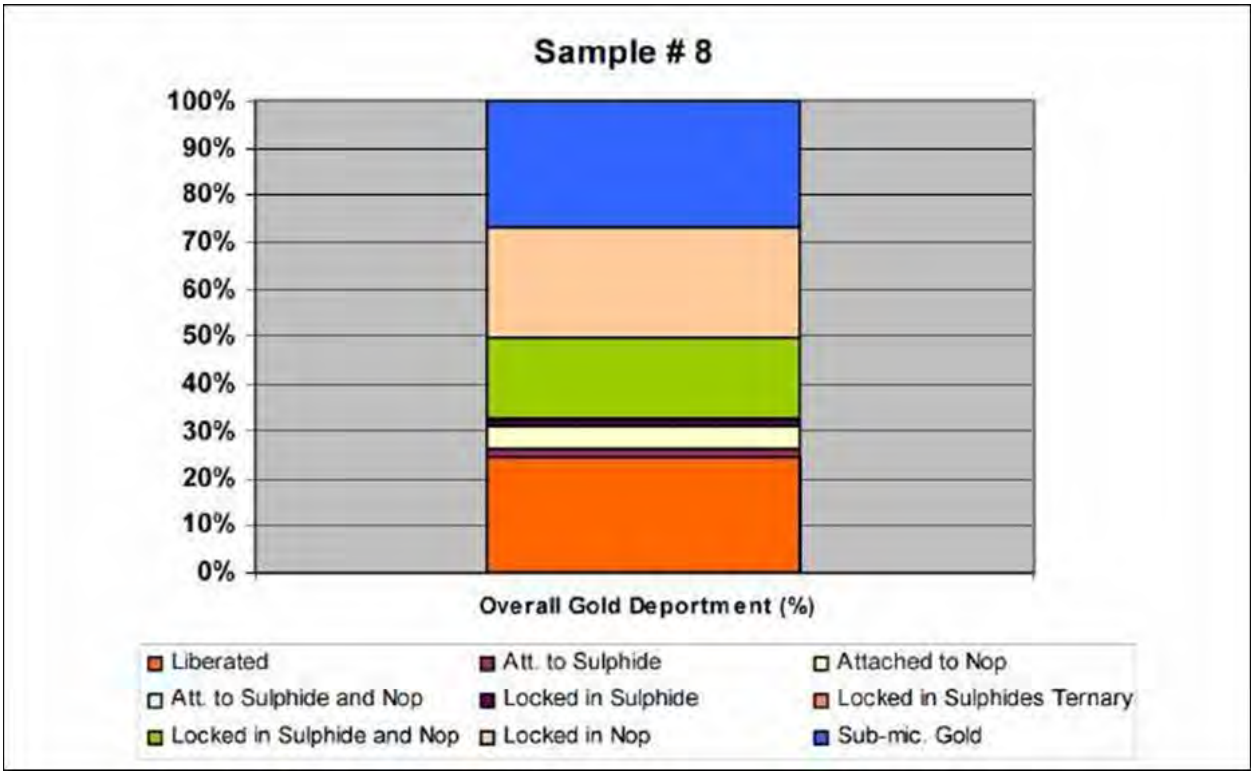
Using a comprehensive analysis approach, and employing methods that included XRD, SEM-EDS, and optical microscope, SGS studied Au deportment on a subsample of Sample #8 stage-pulverized to a P_{80} of 150 µm. The study included microscopic and sub microscopic examination. SGS observed that the major Au mineral is native Au in this sample, suggesting that gravity recovery may be considered in the process flowsheet. The other Au minerals include electrum, maldonite (Au_2Bi), petzite (Ag_3AuTe_2) and Hessite (Ag_2Te). The relative abundance of Au minerals is detailed in Figure 13-2.

As part of a sub microscopic study, SGS evaluated a number of minerals for solid-solution and colloidal Au content using a secondary ion mass spectrometer (SIMS). Figure 13-3 plots the detailed Au deportment in the tested sample.



Source: SGS, May 2011

Figure 13-2: Abundance of gold minerals in Sample 8



Source: SGS, May 2011

Figure 13-3: Overall gold deportment in Sample 8

13.4 Comminution Tests

Comminution tests in testwork program 2011 included SMC/JK drop-weight, Bond ball mill work index (BWi), Bond rod mill work index (RWi), abrasion index (Ai) and HPGR test. No CWi tests were performed so far on any sample of the Toroparu Deposit.

13.4.1 Grindability

Table 13-13 presents the SMC test results, conducted on samples #4,5,6,7, and Master Composite (Sample #8) from 2011 testwork program. As can be seen the values of A x b, ta and DWi are 22.6, 0.22 and 12 kWh/t for the Sample #8, respectively which all indicate that the material is in the category of a hard-mineralized material compared to the accumulated values in the JK Tech DW database.

Table 13-13: Summary of JK Tech/SMC Data (2011)

Sample #	RelativeDensity	JK Parameters		DW_I
		A x b	t_a	(kWh/t)
Sample 4	2.74	22.0	0.21	12.6
Sample 5	2.76	23.0	0.22	11.8
Sample 6	2.77	24.7	0.23	11.3
Sample 7	2.76	24.2	0.23	11.3
Sample 8	2.74	22.6	0.22	12.0

Source: JK Tech, 2011

The Bond ball mill work indices (BWi) are listed in Table 13-14 for three saprolite samples (samples 1, 2, and 3) ground to 100 mesh. BWi was also adjusted to include fine material that bypassed the test procedure. The Master Composite (Sample #8) was ground to both 100 and 200 mesh. Bond rod mill work index was only performed on the Master Composite sample. BWi for two mesh sizes of 100 and 200 mesh for the Master Composite (Sample #8) are 18.2 and 17.7 kWh/t while the RWi for the same sample is 19.3 kWh/t. The high values of BWi and RWi confirm the hardness of the mineralized material.

Table 13-14: Grindability Data

Sample #	RW_i (kWh/t)	BW_i 100 Mesh (kWh/t)	BW_i (adjusted) 100 Mesh (kWh/t)	BW_i 200 Mesh (kWh/t)
Sample 1		7.5	1.3	
Sample 2		11.5	2.9	
Sample 3		8.9	2.0	
Sample 8	19.3	18.2	18.2	17.7

Source: JK Tech, 2011

13.4.2 Abrasion Index

SGS performed a bond abrasion index (Ai) test on a composite Sample 8, with a reported result of 0.294 g placing the sample into the abrasive range.

13.4.3 HPGR Testing

Due to the relative hardness of the material, HPGR in the third stage of crushing was tested as an alternative to a conventional crushing and semi-autogenous grinding (SAG) milling circuit. One HPGR test program was completed in June 2011. This work was carried out at SGS Lakefield, using a Labwal unit, on Master Composite material (Sample #8).

The most relevant test parameters required for equipment sizing were determined, including:

- Specific throughput rate;
- Specific grinding force; and
- Specific energy input versus specific grinding force.

The test program included single-pass tests at three different pressure settings in order to determine the optimum operating parameters for the test apparatus. The test program also incorporated locked cycle testing in order to simulate the product size distributions to be expected in an industrial sized HPGR circuit. The results of the locked cycle testwork are summarized in Table 13-15.

Table 13-15: Summary of HPGR Test Findings

Description	Unit	Value
Wet Bulk Density	kg/L	1.75
Feed Particle Size, F_{80}	mm	10
Product Particle Size, P_{80}	mm	2.3
Pressure of Operation	bar	60
Moisture	(% H_2O)	3.6
Dry Net Throughput	t/h	1.5
Circulating Load	%	72.4
Gross Specific Energy Requirement	kWh/t	3.70
Net Specific Energy Requirement	kWh/t	3.06
Specific Grinding Force	N/mm^2	3.01
Specific Throughput	$t*s/m^3*h-(m_f)$	220
Specific Throughput Rate	$t*s/m^3*h-(m_c)$	196
Ratio m_c/m_f		0.89

Source: SGS, 2011

The results indicate that the sample material is amenable to the HPGR process.

13.5 Metallurgical Tests

Focusing on the production of two on site final products (Cu concentrate and Au doré) metallurgical tests primarily consisted of:

- Gravity separation testwork;
- Flotation testwork; and
- Cyanidation leaching.

Some environmental testwork including SPLP (Synthetic Precipitation Leaching), Acid Base Accounting (ABA) and Net Acid Generation (ABA) and some cyanide destruction tests on cyanide tailing products were also carried out at a preliminary level.

13.6 Metallurgical Test Work Program 2009

An initial metallurgical scoping test program was conducted on the saprolite and hardrock mineralized material samples from the Toroparu Deposit. The goal of this program was to scope the amenability of Toroparu Deposit mineralized material to typical Au extraction and Cu recovery methods. In addition, baseline environmental and batch cyanide destruction tests were conducted to identify potential environmental liabilities associated with the conditions under consideration.

The saprolite tailing mixture sample assayed 0.6 g/t Au. The hardrock core composite assayed 1.43 g/t Au and 0.17% Cu.

The rougher flotation scoping tests on the hardrock composite showed that the Cu and Au recoveries in the rougher concentrate were approximately 97% and 93%, respectively. The mass pull was 2.5% to 3.6%. These results show that the Cu and Au in the hardrock mineralized material were effectively recovered by flotation.

The 72-hour cyanidation and 48-hour CIL tests on the rougher flotation tailing from the hardrock composite showed that 70% to 74% of the Au could be extracted. Conventional cyanidation conditions were applied in the tests. The overall Au recovery in the flotation rougher concentrate and cyanidation of flotation tailing was approximately 98%. A 72-hour cyanidation test on the Cu rougher flotation concentrate indicated that approximately 68% of Au was leached. The 72-hour cyanidation and 48-hour CIL tests on the saprolite gravity tailing mixture revealed that >95% of the Au was extracted using conventional cyanidation conditions, with low cyanide consumption. The results indicate that the saprolite tailings mixture responded well to cyanidation.

A basic environmental test program characterized three waste samples:

- A blend of two waste rocks (Blended Waste Rock);
- A CIL residue of hardrock flotation tailings (F2 Ro Tails); and
- A saprolite tailing CIL residue (CIL-6 Residue).

Whole rock analyses determined that the Toroparu Deposit waste products were comprised primarily of silicates with moderate amounts of aluminum and iron. Minor amounts of calcium and sodium were also evident in the Blended Waste Rock and F2 Ro Tails samples; however, only trace amounts of these elements were reported in the CIL-6 Residue sample.

Modified ABA tests clearly indicated the significant neutralization capacity of the Blended Waste Rock and F2 Ro tailings samples and suggested that these samples have the potential for acid consumption. The non acid generating nature of these samples was confirmed during ABA testing.

ABA testing of the CIL-6 Residue sample suggested uncertainty with regard to the acid generation potential. The low sulphide content (<0.01%) reported indicates that acid generation is highly unlikely to occur from this sample. ABA testing of the CIL-6 Residue sample reported no net acidity generated and an alkaline final pH value corroborating the highly unlikely acid generating designation.

To explore the amenability of the CIL discharges to detoxification, two batch cyanide destruction tests were carried out using the Air/SO2 method. One was conducted on the CIL barren solution of the hardrock rougher flotation tailing (test CIL-5), and another was conducted on the CIL barren solution of the gravity tailing mixture (test CIL-6). Both of the tests showed that the CN_{WAD} and CNT contents were lowered to <0.1 ppm and <0.4 ppm, respectively. The retention times were equal or less than 90 minutes. The reagent consumptions for hardrock CIL solution was 4.99 g equivalent SO2 and 3.30 g hydrated lime per gram of cyanide in the feed. The reagent requirement for the saprolite tailing mixture CIL solution was 5.14 g equivalent SO_2 and 3.57 g hydrated lime per gram of cyanide in the feed.

More tests were conducted in future test programs to optimize the retention time and minimize the reagent consumptions.

13.7 Metallurgical Test Work Program 2011

The testwork conducted by SGS Lakefield in 2011 using Sample #8 includes three phases, Phase 1, Phase 2, and Phase 2 Extension; all phases included gravity separation, flotation, and cyanidation testwork.

13.7.1 Gravity Separation

In 2011, SGS Lakefield conducted gravity concentration tests on composite Sample #8 in all three phases using a Knelson MD-3 Concentrator to produce a concentrate that was further upgraded using a Mozley C800 Laboratory Separator.

In Phase 1, two gravity recovery tests were completed with a target grind size of 48 mesh (300 µm) and 100 mesh (150 µm) which resulted in 32.5% and 36.5% Au recovery at the grind P_{80} of 259 µm and 151 µm respectively. In Phase 2, four tests were conducted with 20 kg feed charges and each feed sample was ground to a different size, with target P_{80}'s of 50, 75, 125 and 175 µm. The result shows that Au recovery changes in the range of 13.4% to 52.1% for the selected grind sizes with 13.4% for a grind size of 175 µm and 52.1% for a grind size of 75 µm. In the last phase (Phase 2 Extension) two tests using 2 kg charges, ground to P_{80} 228 µm and 149 µm were completed, which resulted in 47.7% and 43.4% Au recovery, respectively. The test results in all three phases show that there is considerable deviation in Au recovery between the different grind sizes. However, the Phase 2 Extension tests results indicate that an average Au recovery of 45% with the grade of 130 g/t in coarser sizes (150 to 228 µm) is achievable.

The results of the three phases have been summarized in Table 13-16.

Table 13-16: Gravity Separation Results Summary for Phase 1, Phase 2, Phase 2 Extension

Gravity Test No.	Feed Size P_{80}, µm	Feed Weight kg	Product	Mass %	Assays Au, g/t	Distribution %
Phase1						
MC-04	300	2	Mozley Concentrate	0.07	282	32.5
			Knelson/Mozley Tailing	99.93	0.39	67.5
MC-05	150	2	Mozley Concentrate	0.12	189	36.5
			Knelson/Mozley Tailing	99.88	0.4	63.5
Phase 2						
	50	20	Mozley Concentrate	0.03	652	31.0
			Knelson/Mozley Tailing	99.97	0.39	69.0
	75	20	Mozley Concentrate	0.04	838	52.1
			Knelson/Mozley Tailing	99.96	0.33	47.9
	125	20	Mozley Concentrate	0.03	517	24.8
			Knelson/Mozley Tailing	99.97	0.44	75.2
	175	20	Mozley Concentrate	0.03	370	13.4
			Knelson/Mozley Tailing	99.97	0.67	86.6
Phase2 Extension						
G-51	228	2	Mozley Concentrate	0.24	122	47.7
			Knelson/Mozley Tailing	99.8	0.33 *	52.3
G-52	149	2	Mozley Concentrate	0.20	145	43.4
			Knelson/Mozley Tailing	99.8	0.37 *	56.6

Source: SGS, 2011. Knelson/Mozley tailings is calculated from cyanidation test

The Company suspected that the variability in Au recovery between the different grind sizes in the Phase 2 testwork was due to overloading of the test equipment and had a set of the same samples prepared for independent testing at Resource Development Inc. (RDi).

RDi performed gravity concentration tests on four, 1 kg charges of Sample# 8, (prepared by SGS). The four charges were stage ground to four nominal grind sizes of P_{80} 50, 75, 125, and 175 µm. The detailed results of the gravity concentration testwork can be found in

Table 13-17.

Table 13-17: Gravity Separation Results Summary for RDI Testwork

Gravity Test No.	Feed Size P_{80}, µm	Feed Weight, g	Product	Mass %	Assay Au, g/t	Distribution %
1	175	955.6	Gemeni Concentrate	3.1	9.46	48
			Knelson/Gemeni Tailing	96.9	0.30	52
			Head (Calculated)		0.60	100
2	125	970.5	Gemeni Concentrate	2.1	16.05	54.6
			Knelson/Gemeni Tailing	97.9	0.28	45.4
			Head (Calculated)		0.62	100
3	75	972.8	Gemeni Concentrate	1.9	13.72	50.7
			Knelson/Gemeni Tailing	98.1	0.25	49.3
			Head (Calculated)		0.52	100
4	50	979.5	Gemeni Concentrate	4.6	7.2	36.6
			Knelson/Gemeni Tailing	95.4	0.54	63.4
			Head (Calculated)		0.90	100

Source: SGS, 2011

The tests indicate a range of recoveries of 36.6% to 54.6% across the grind sizes tested. These results are consistent with both the Phase 1 and Phase 2 extension results and indicate that a 30% to 50% gravity recovery of Au may be possible at grind sizes of approximately 150 µm.

13.7.2 Flotation Testwork

Each of the test programs conducted by SGS Lakefield on Master Composite (Sample #8) indicated that Cu and Au were effectively recovered in flotation testwork.

Phase 1

Effect of the Primary Grind Size

In Phase 1, three flotation tests including one rougher at P_{80} 200 mesh and two cleaner at P_{80} targets of 100 and 200 mesh were conducted. Cytec Aerophine promoter 3418A and Potassium Amyl Xanthate (PAX) as collectors were used in all three tests.

Flotation at the targeted 200 mesh (75 µm) grind size achieved rougher recoveries of 97.4% Cu and 92.6% Au at a grind P_{80} of 61 µm. The cleaner tests performed at the similar grind size (58 µm) achieved similar rougher recoveries of 97.2% Cu and 92.0% Au and a final concentrate grade of 29.6% Cu and 161 g/t Au at a recovery of 67.0% Cu and 70.4% Au. Another cleaner test was conducted targeting a 100 mesh (150 µm) grind and 182 µm was achieved. The rougher recovery was 88.9% Cu and 88.0% Au and final concentrate grade of 32.2% Cu and 136 g/t Au at recovery of 60.5% Cu and 49.1% Au. The results are summarized in Table 13-18.

Table 13-18: Flotation Test Results Summary for Phase 1

Test ID	Product	Primary Grind Size P_{80} (µm)	Regrind P_{80} (µm)	Wt%	Assays (%, g/t)		Distribution (%)	
					Cu	Au	Cu	Au
MC-01	Rougher Conc	61		6.1	2.29	11.5	97.4	92.6
	Rougher Tail			93.9	0.004	0.06	2.6	7.4
	Head (calc.)			100	0.14	0.76	100	100
MC-02	3rd Clnr Conc	182	~10	0.26	32.2	136	60.5	49.1
	Rougher Conc			4.02	3.07	15.8	88.9	88.0
	Rougher Tail			96.0	0.016	0.09	11.1	12.0
	Head (calc.)			100	0.14	0.72	100	100
MC-03	3rd Clnr Conc	58	~10	0.31	29.6	161	67.0	70.4
	Rougher Conc			5.05	2.66	13.00	97.2	92.0
	Rougher Tail			95.0	0.004	0.06	2.76	7.99
	Head (calc.)			100	0.14	0.71	100	100

Source: SGS, 2011

The result shows that Au and Cu recovery in rougher improves with an increase in primary grinding size fineness.

Phase 2

The flotation test work in Phase 2 conducted on two separate flowsheet processes for recovering Au and Cu at four primary grind size P_{80} target 50, 75, 125, and 175 µm.

The first process included a matrix of 16 rougher flotation tests conducted on gravity tails to explore the effect of grind and reagent mix on the metallurgical performance of composite Sample# 8.

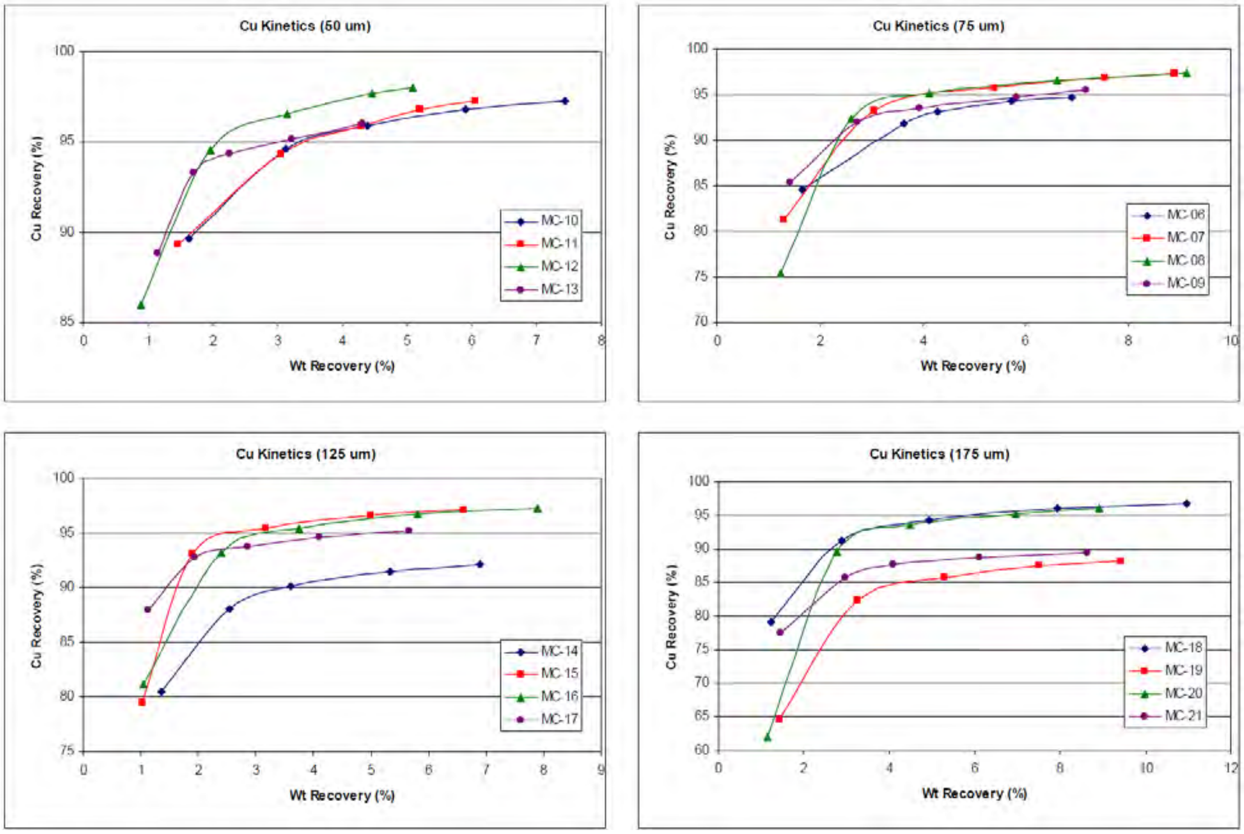
The second process included a total of eight rougher/cleaner flotation tests performed on the cyanide leach residues produced from the direct cyanidation of the gravity tailings.

Rougher Flotation Tests on Gravity Tailings

Tailing products from the gravity separation tests in Phase 2 were used as feed for the rougher flotation tests MC-06 to MC-21, the results have been summarized in Table 13-19. Reagent schemes A, B, C, and D have been tested for each grind size. The reagents include A3418A, PAX, AF208, A407, H2SO4 and CuSO4.

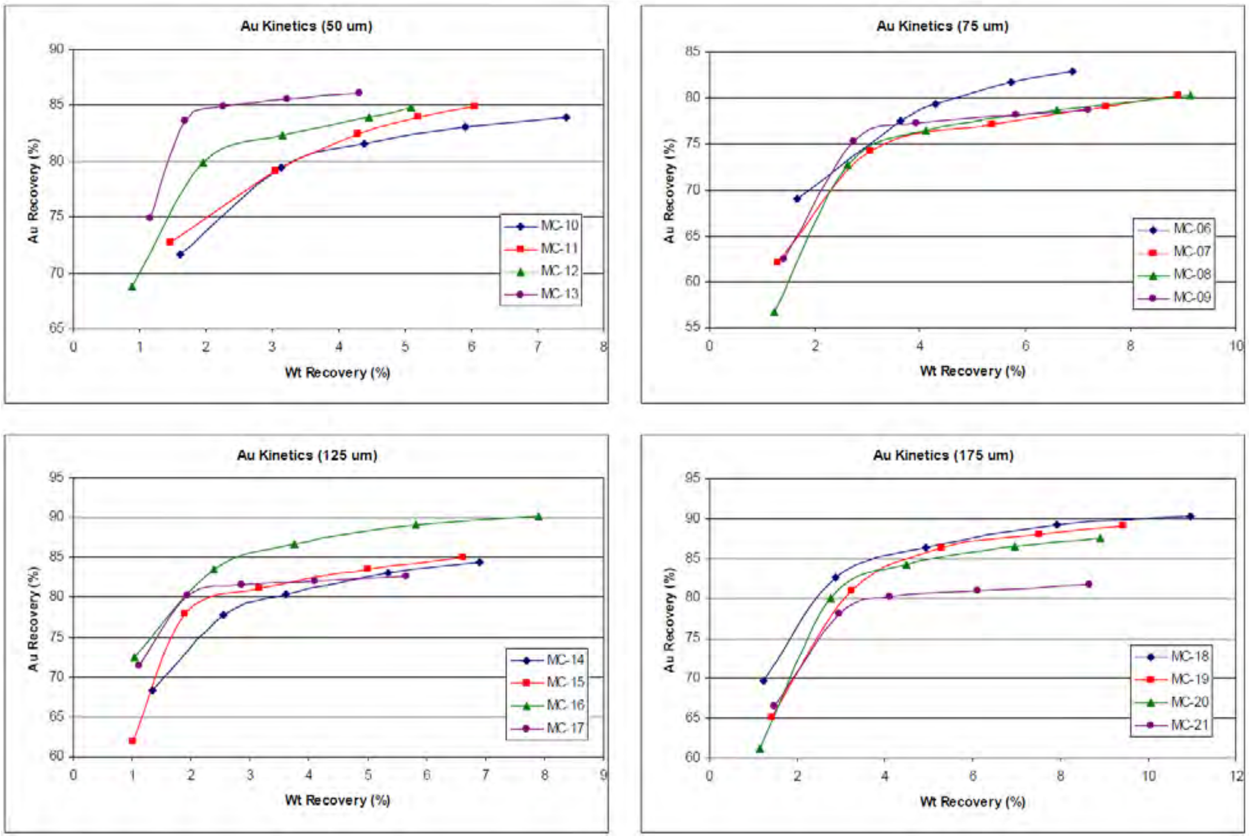
Figure 13-4 and Figure 13-5 show the effect of the reagent scheme on flotation kinetics of Cu and Au for each primary grind size. The results show that Cu and Au were effectively recovered in these flotation tests and the flotation kinetics responses for both Cu and Au were quite fast for the first two minutes, and after ten minutes were slow. In general mass recoveries in the range of 5% to 11% were achieved after 30 minutes of flotation. Flotation kinetics indicated a faster flotation of Cu than Au.

In general, higher recoveries of Cu can be achieved in finer sizes while the higher recoveries were observed for Au at coarser primary grind sizes especially using reagent schemes A, B, and C.



Source: SGS, October 2012

Figure 13-4: Cu Flotation kinetics – comparison of reagent scheme for each feed P_{80}



Source: SGS, October 2012

Figure 13-5: Gold flotation kinetics comparison of reagent scheme for each feed P_{80}

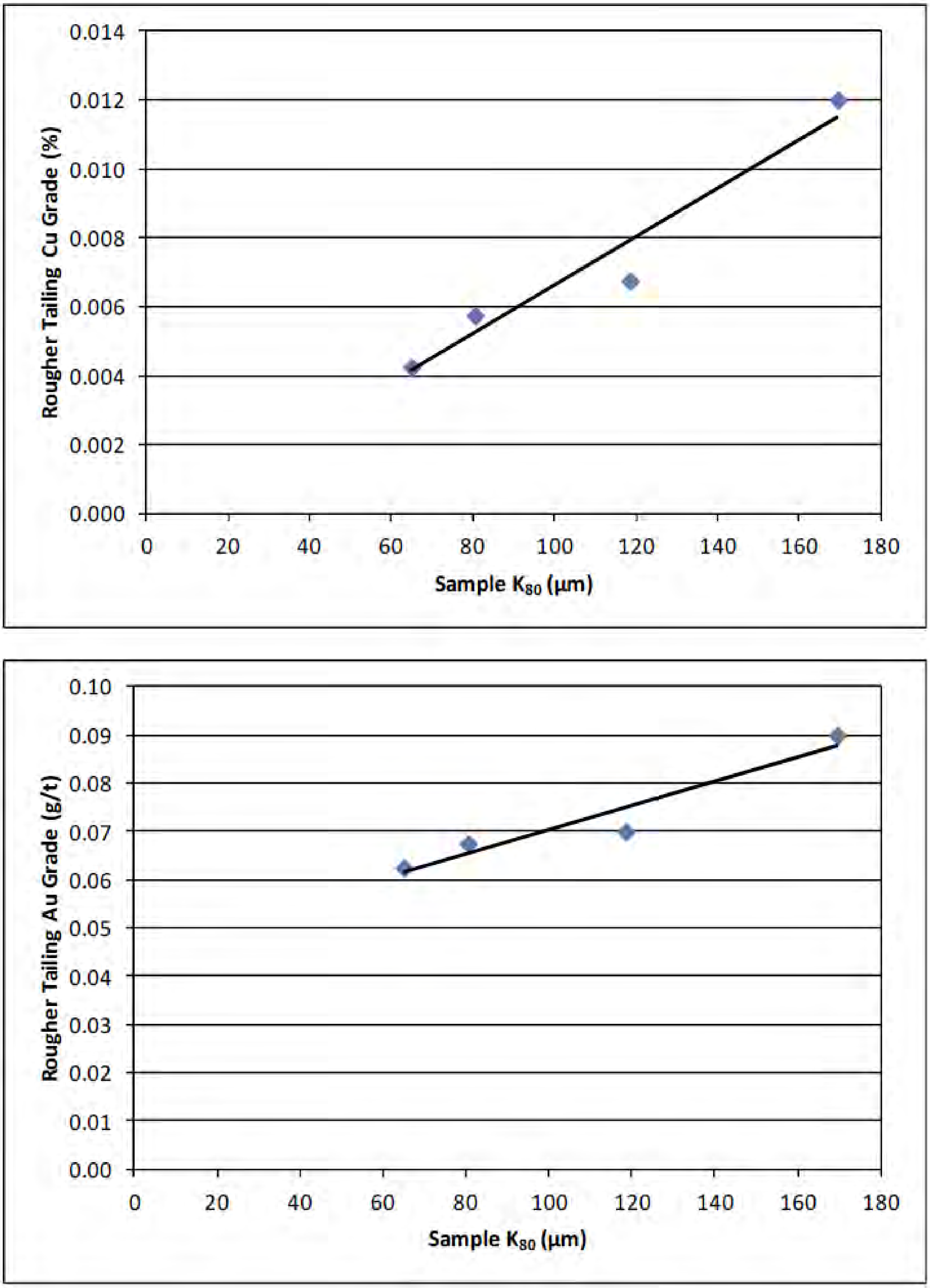
Table 13-19: Gravity Tail Rougher Flotation Test Conditions

Test ID	Reagent Scheme	Product	Grind Size P_{80} (µm)	Wt%	Assays (%, g/t)		Distribution (%)	
					Cu	Au	Cu	Au
MC-06	A	Rougher Conc 1-5	75	6.90	1.97	3.93	94.8	2.9
		Rougher Tail		93.1	0.008	0.06	5.2	17.1
		Head (calc.)		100	0.14	0.33	100	100
MC-07	B	Rougher Conc 1-5	75	8.91	1.53	2.91	97.4	80.3
		Rougher Tail		91.1	0.004	0.07	2.60	19.7
		Head (calc.)		100	0.14	0.32	100	100
MC-08	C	Rougher Conc 1-5	75	9.14	1.49	2.83	97.4	80.3
		Rougher Tail		90.9	0.004	0.07	2.61	19.7
		Head (calc.)		100	0.14	0.32	100	100
MC-09	D	Rougher Conc 1-5	75	7.18	1.92	3.34	95.5	78.7
		Rougher Tail		92.8	0.007	0.07	4.50	21.3
		Head (calc.)		100	0.14	0.3	100	100
MC-10	A	Rougher Conc 1-5	50	7.44	1.76	4.58	97.2	84.0
		Rougher Tail		92.6	0.004	0.07	2.75	16.0
		Head (calc.)		100	0.13	0.41	100	100
MC-11	B	Rougher Conc 1-5	50	6.06	2.20	5.24	97.3	84.9
		Rougher Tail		93.9	0.004	0.06	2.74	15.1
		Head (calc.)		100	0.14	0.37	100	100

Test ID	Reagent Scheme	Product	Grind Size P_{80} (µm)	Wt%	Assays (%, g/t)		Distribution (%)	
					Cu	Au	Cu	Au
MC-12	C	Rougher Conc 1-5	50	5.10	2.75	6.22	98.0	84.8
		Rougher Tail		94.9	0.003	0.06	1.99	15.2
		Head (calc.)		100	0.14	0.37	100	100
MC-13	D	Rougher Conc 1-5	50	4.32	3.21	8.26	96.0	86.1
		Rougher Tail		95.7	0.006	0.06	3.98	13.9
		Head (calc.)		100	0.14	0.41	100	100
MC-14	A	Rougher Conc 1-5	125	6.90	1.88	5.13	92.1	84.5
		Rougher Tail		93.1	0.012	0.07	7.93	15.5
		Head (calc.)		100	0.14	0.42	100	100
MC-15	B	Rougher Conc 1-5	125	6.61	1.91	5.60	97.1	85.0
		Rougher Tail		93.4	0.004	0.07	2.87	15.0
		Head (calc.)		100	0.13	0.44	100	100
MC-16	C	Rougher Conc 1-5	125	7.89	1.64	5.35	97.2	90.2
		Rougher Tail		92.1	0.004	0.05	2.76	9.83
		Head (calc.)		100	0.13	0.47	100	100
MC-17	D	Rougher Conc 1-5	125	5.66	2.33	7.12	95.2	82.6
		Rougher Tail		94.3	0.007	0.09	4.77	17.4
		Head (calc.)		100	0.14	0.49	100	100
MC-18	A	Rougher Conc 1-5	175	11.0	1.24	5.33	96.8	90.4
		Rougher Tail		89.0	0.005	0.07	3.19	9.65
		Head (calc.)		100	1.14	0.65	100	100
MC-19	B	Rougher Conc 1-5	175	9.42	1.43	5.51	88.2	89.10
		Rougher Tail		90.6	0.020	0.07	11.8	10.9
		Head (calc.)		100	0.15	0.58	100	100
MC-20	C	Rougher Conc 1-5	175	8.91	1.49	6.48	96.1	87.6
		Rougher Tail		91.1	0.006	0.09	3.94	12.4
		Head (calc.)		100	0.14	0.66	100	100
MC-21	D	Rougher Conc1-5	175	8.64	1.53	6.15	89.5	81.7
		Rougher Tail		91.4	0.017	0.13	10.5	18.3
		Head (calc.)		100	0.15	0.65	100	100

Source: SGS, 2012

The effect of primary grind size on recovery of Cu and Au can be also evaluated by the amount of Cu and Au that has been lost in the rougher tailings. Figure 13-6 presents the average rougher tailings grade over the four tests at each primary grind size against P_{80}.



Source: SGS, Oct 2012

Figure 13-6: Effect of sample P_{80} on recovery of Cu and Au

The results show fairly consistent tailings grades with the exception of the coarsest grind 175 µm, in which tailings are notably higher. Therefore, a P_{80} in the range of 120-150 µm would be a suitable primary grind size for the rougher flotation stage.

Flotation of Cyanide Leach Residues

The testwork program involved flotation of the residual Cu minerals from the Cyanide destruction (CND) pulp following direct cyanide leach of the gravity tailing. The CND process was conducted on a CN_{WAD} content of 0.5 and 2 ppm. Flotation tests were performed for each of the CNWAD levels at each grind size for a total of eight tests (MC-23 to MC-30). Primarily collector A3418A and PAX were used. However, using a weaker collector, isopropyl xanthate (SIPX), along with a short five minute regrind and the addition of activators, sodium hydrosulfide (NaHS) and Cu sulphate ($CuSO_4$), improved the metallurgical response of the pulp following cyanide destruction.

The results show that the flotation response was acceptable with Cu rougher concentrate recoveries ranging from 60% to 80%. Au recoveries were poor, but this was an expected result due to over 90% of the Au having already been leached. Remaining Au was likely unrecoverable; therefore, the focus of these tests was on Cu recovery.

The impact of CN_{WAD} content was negligible for the 50 and 75 µm P_{80} grind sizes, while for coarser grinds the higher CN_{WAD} content caused a substantial decrease in performance as CN behaves as a depressant of Cu flotation. It is expected that the finer grinds and lower CN_{WAD} contents would show improved performance.

Phase 2 Extension

Three series of open circuit rougher/cleaner flotation tests where completed (MC-31, MC-32 to MC- 35, and MC-37 to MC-40). The cleaner concentrate and cleaner tailing of these tests were used in cyanidation tests. The results are summarized in

Table 13-20 for test MC-31 to MC-35 and

Table 13-21 for test MC-37 to MC-40.

Table 13-20: Results of Flotation Test MC-31 to MC-35

Test ID	Product	GrindSize P_{80} (µm)	Wt%	Assays (%, g/t)		Distribution %	
				Cu	Au	Cu	Au
MC-31	2nd Clnr Conc	138	0.66	16.0	73.1	79.1	64.8
	1st Clnr Conc		1.71	6.80	31.1	87.3	71.5
	1st Clnr Tail		6.14	0.11	1.81	5.13	14.9
	Ro Conc		7.85	1.57	8.20	92.4	86.4
	Ro Tail		92.1	0.011	0.11	7.6	13.6
	Head (Calc)		100	0.13	0.75	100	100
MC-32	2nd Clnr Conc	146	0.70	17.1	72.0	80	63.8
	1st Clnr Conc		1.85	7.17	47.6	89.3	84.8
	1st Clnr Tail		6.29	0.078	0.47	3.29	3.71
	Ro Conc		8.14	1.69	11.2	92.6	87.6
	Ro Tail		91.9	0.012	0.14	7.4	12.4
	Head (Calc)		100	0.15	1.04	100	100
MC-33	2nd Clnr Conc	174	0.70	17.1	72.0	77.8	76.3
	1st Clnr Conc		1.72	7.82	32.6	87.2	84.4
	1st Clnr Tail		6.32	0.078	0.47	3.2	4.44
	Ro Conc		8.04	1.73	7.32	90.4	88.9
	Ro Tail		92.0	0.016	0.08	9.56	11.1
	Head (Calc)		100	0.15	0.66	100	100
MC-34	2nd Clnr Conc	199	0.69	17.1	72.0	82.1	74.3
	1st Clnr Conc		1.39	9.04	38.1	87.6	79.1
	1st Clnr Tail		6.23	0.078	0.47	3.38	4.32
	Ro Conc		7.62	1.72	7.34	91	83.5
	Ro Tail		92.4	0.014	0.12	9	16.5
	Head (Calc)		100	0.14	0.67	100	100
MC-35	2nd Clnr Conc	222	0.71	17.1	72.0	74.8	71.7
	1st Clnr Conc		1.44	9.81	40.8	87.8	82.8
	1st Clnr Tail		6.37	0.078	0.47	3.08	4.17
	Ro Conc		7.81	1.88	7.91	90.8	87.0
	Ro Tail		92.2	0.016	0.10	9.15	13.0
	Head (Calc)		100	0.16	0.71	100	100

Source: SGS, 2012

Table 13-21: Results of Flotation Test MC-37 to MC-38

Test ID	Product	Grind Size (µm)		Wt%	Assays (%, g/t)			Distribution %		
		Primary	Regrind		Cu	Au	Ag	Cu	Au	Ag
	Mozley Conc			0.10	8.28	164	170	7.2	33.1	10.6
	Mzly + 2nd Clnr Conc			0.61	16.3	64.0	144	83.3	76.1	52.8
	Mzly + 1st Clnr Conc			1.43	7.46	28.6	68.0	88.8	78.9	57.8
	Mzly + 1st Cl SC Conc			2.42	4.53	17.3	42.3	91.4	80.8	61
	1st Clnr Tail	130	18	7.47	0.046	0.35	2.00	2.3	4.4	7.9
MC-37	Mozley + Ro Conc			13.5	0.85	3.38	9.23	95.7	88.3	74.2
	Ro Tail			86.5	0.006	0.07	0.50	4.3	11.7	25.8
	Head (Calc)			100	0.12	0.52	1.68	100	100	100
	Mozley Conc			0.05	7.28	400	372	2.9	37.5	11.3
	Mzly + 2nd Clnr Conc			0.63	11.2	54.0	117	56.8	63.5	44.7
	Mzly + 1st Clnr Conc			2.23	4.53	18.1	45.8	81.8	75.7	62.2
	Mzly + 1st Cl SC Conc			3.51	2.99	11.9	31.1	85.0	78.1	66.4
	1st Clnr Tail	202	20	7.30	0.11	0.53	1.30	5.6	6.7	5.2
MC-38	Mozley + Ro Conc			12.3	0.93	3.76	9.78	92.9	86.9	73.3
	Ro Tail			87.7	0.010	0.08	0.50	7.1	13.1	26.7
	Head (Calc)			100	0.12	0.53	1.65	100	100	100
	Mozley Conc			0.01	6.45	960	382	0.7	22.3	3.0
	Mzly + 2nd Clnr Conc			0.48	14.5	53.7	141	61.5	47.5	42.3
	Mzly + 1st Clnr Conc			1.68	5.53	20.9	57.2	82.4	64.9	60.0
	Mzly + 1st Cl SC Conc			2.61	3.7	13.8	39	85.7	66.8	63.6
	1st Clnr Tail	231	19	7.43	0.053	0.38	1.4	2.8	4.7	5.6
MC-39	Mozley + Ro Conc			13.2	0.77	3.04	8.84	90.7	74.2	72.8
	Ro Tail			86.8	0.012	0.16	0.5	9.3	25.8	27.2
	Head (Calc)			100	0.11	0.54	1.6	100	100	100
	Mozley Conc			0.07	5.94	252	195	3.2	31.8	7.8
	Mzly + 2nd Clnr Conc			0.68	13.7	52.6	133	71.7	65.3	52.6
	Mzly + 1st Clnr Conc			2.27	4.84	17.3	48.6	84.2	71.7	64
	Mzly + 1st Cl SC Conc			3.27	3.49	12.4	35.8	87.7	74.2	68
	1st Clnr Scav Tail	283	19	7.38	0.057	0.47	1.20	2.7	5.8	4.5
MC-40	Mozley + Ro Conc			13.1	0.91	3.45	9.82	92.0	82.6	74.8
	Ro Tail			86.9	0.012	0.11	0.50	8.0	17.4	25.2
	Head (Calc)			100	0.13	0.55	1.72	100	100	100

Source: SGS, 2012

AF-208 and PAX were used as collectors. Tests MC-32 to MC-35 were performed on four different primary grind sizes of P_{80} 125,175, 210, and 250 µm and tests MC-37 to MC-40 were also performed on four different primary grind sizes of P_{80} 150, 210, 250, and 300 µm with being regrind to P_{80} 18 µm before the cleaning stage.

The results of the test series MC-32 to MC-35 demonstrated an overall recovery drop for both Cu and Au with increasing grade. Cu cleaner recoveries were in the range of 74.8 to 82.1 with a grade of 17.1% Cu while Au recoveries were in the range of 63.8 to 76.3 with a grade of 72 g/t in the cleaner stage.

The results of the test series MC-37 to MC-40 showed that the performance of the cleaner circuit did not appear to be consistent with wide variation in grade-recovery for the different grind sizes. Recovery of the rougher concentrate portion appeared to be consistent and clearly demonstrated that the finer grind of P_{80} 150 µm outperformed the coarser size grinds. This was expected from the mineralogy examination of the feed that indicated drastically improved liberation of Cu minerals at P_{80} 150 µm.

Overall, the test results of series MC-37 to MC-40 showed that the Cu and Ag recoveries were more consistent than Au which probably can be associated with inconsistent results from the Mozley concentration step before flotation. The best Cu, Au and Ag recoveries of 83.3%, 76.1% and 52.8% respectively were obtained in the grind size of P_{80} 150 µm.

13.7.3 Cyanide Leaching

The cyanide leaching in Phase 1 was presented in two separate flowsheet processes, first was the cyanidation of whole ore (P_{80} target of 150 µm and 75 µm) and the second was rougher concentrate cyanidation.

The cyanidation testwork involved three whole mineralized material tests, one, at P_{80} 150 µm and two at P_{80} 75 µm. One of the P_{80} 75 µm leaches was performed as a CIL test while the other did not have any carbon added. Following a 48-hour leach residence time, extractions of 90.0% Au at feed P_{80} of 160 µm, and 89.1% Au at feed P_{80} of 102 µm were achieved. The CIL had a total extraction of 91.1% Au, with 88.7% contained in the carbon and 2.4% remaining in the barren leach solution. The cyanidation testing also included leaching of rougher concentrate in two tests where one was tested as received while the other was reground. Extraction of 96.4% and 97.8% Au was achieved from the as received and reground samples respectively. Cu extraction and cyanide consumption increased from 18.9% Cu and 14.7 g of NaCN for the as received sample test to 29.6% Cu and 27.4 g of NaCN for the reground sample. It appears that regrinding of the sample may have led to increased leaching of Cu resulting in a higher Cu concentration in solution.

The cyanide leaching in Phase 2 included leaching of four separate products. The first set of leach tests was performed on the gravity tailings resulting from four target grinds; P_{80}'s of 50, 75, 125 and 175 µm. Following direct cyanidation of the gravity tailing, the leach residues were combined and then split for performing two CND tests that were completed to a weak acid dissociable (WAD) content of 0.5 and

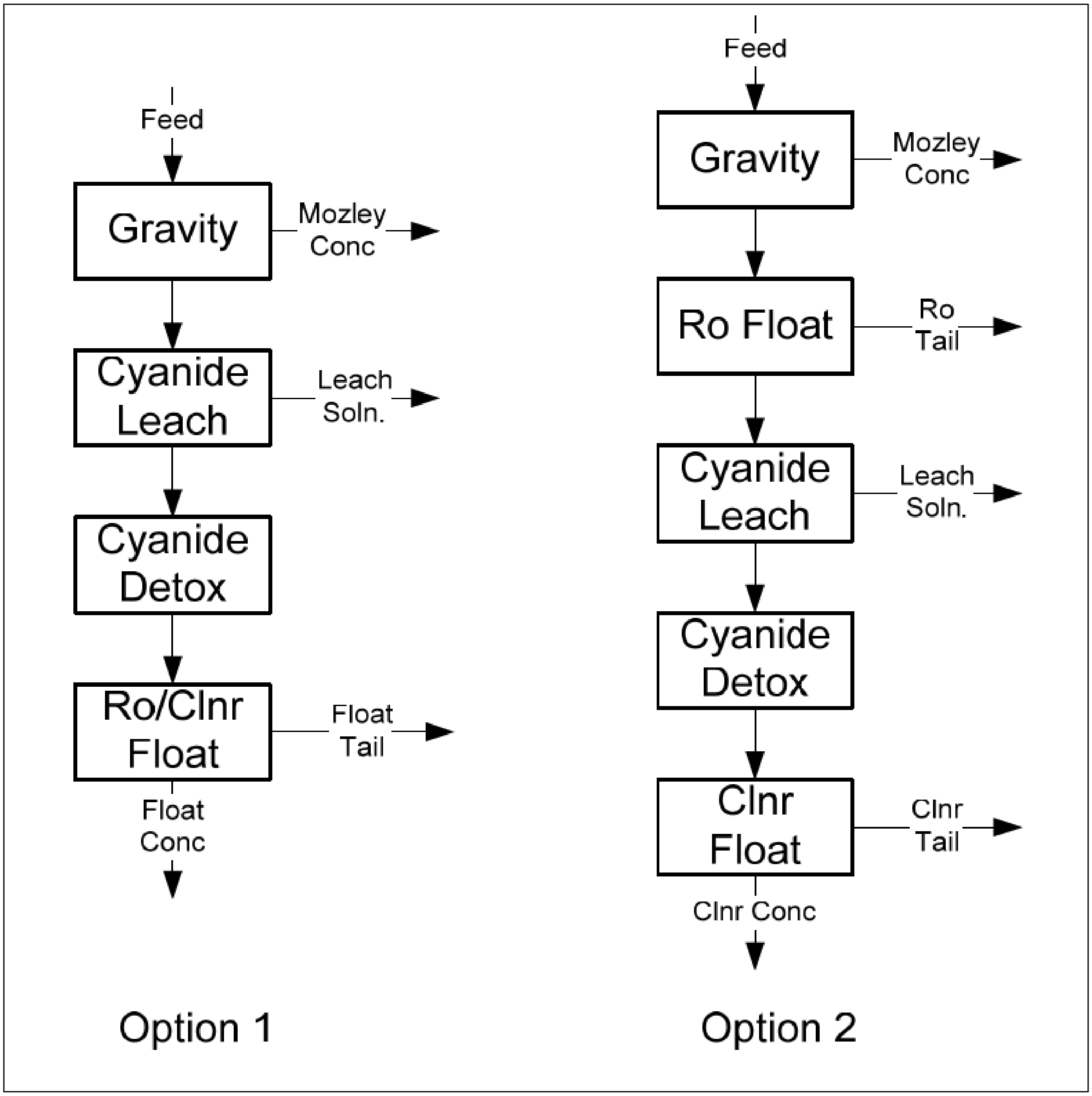
2.0 ppm. The CND test residues were then tested for recovery of Cu using flotation.

The second set of leach tests was performed on rougher tailings following the rougher flotation of each size campaign.

The third set of leach tests was performed on the rougher concentrates from the 75 µm campaign. The rougher concentrate was split into thirds and the effect of regrind was evaluated. The three leach residues were then combined, and a CND test was completed to a WAD content of 0.5 ppm after which the recovery of Cu using flotation was again examined.

The fourth set of leach tests was performed on Mozley vanner gravity concentrate from the 75 µm campaign.

The purpose of this highly integrated testwork program was to evaluate the effect of grind size as well as to evaluate the effect of the processing flowsheet on the recovery of Au and Cu. The two conceptual flowsheets being evaluated during this testing program are summarized in Figure 13-7.



Source: SGS, October 2012

Figure 13-7: Conceptual flowsheets

The results of the Phase 2 testwork program for Au and Cu recovery have been summarized in Table 13-22 and the cyanide consumptions for the two options are shown in Table 13-23.

Table 13-22: Results of Phase 2 Testwork Program

Flowsheet ID	Product	Au Rec/Ext (%)							
		50 µm		75 µm		125 µm		175 µm	
		Unit	Overall	Unit	Overall	Unit	Overall	Unit	Overall
Gravity	Gravity Con	53.7	53.7	47.7	47.7	8.7	8.7	0.5	0.5
	Gravity Tail	46.3	46.3	52.3	52.3	91.3	91.3	99.5	99.5
Option 1	Gravity Tail Leach	89.3	41.4	86.5	45.2	85.9	78.5	90.1	89.6
	CND Float	37.1	1.8	52.7	3.7	37.8	4.9	26.6	2.6
Option 2	Gravity Tail Float	85.0	39.3	80.5	42.1	85.6	78.1	87.2	86.8
	Ro Conc Leach			90.6	38.2				
	CND Float			60.1	2.4				
	Ro Tail Leach	61.6	4.3	66.4	6.8	52.5	6.9	43.3	5.5
Total	**Option 1**		**96.9**		**96.7**		**92.0**		**92.7**
	Option 2 W. FT Leach				**95.0**				
	Option 2 W/O FT Leach				**88.2**				
Flowsheet ID	**Product**	**Cu Rec/Ext (%)**							
		50 µm		**75 µm**		**125 µm**		**175 µm**	
		Unit	**Overall**	**Unit**	**Overall**	**Unit**	**Overall**	**Unit**	**Overall**
Gravity	Gravity Con	3.1	3.1	2.0	2.0	0.6	0.6	0.1	0.1
	Gravity Tail	96.9	96.9	98.0	98.0	99.4	99.4	99.9	0.99
Option 1	Gravity Tail Leach	17.7	17.2	18.9	18.5	17.9	17.8	17.2	17.1
	CND Float	76.0	73.7	82.0	65.2	73.6	73.1	64.4	64.4
Option 2	Gravity Tail Float	97.1	94.1	96.3	94.4	95.4	94.8	92.6	92.6
	Ro Conc Leach			22.5	21.2				
	CND Float			90.0	65.8				
	Ro Tail Leach	33.9	0.9	34.1	1.2	32.4	1.5	29.5	2.2
Total	**Option 1**		**73.7**		**85.7**		**73.1**		**64.4**
	Option 2 W. FT Leach				**90.3**				
	Option 2 W/O FT Leach				**89.0**				

Source: SGS, 2012

Table 13-23: Comparison of Cyanide Consumption for Phase 2 Testwork Program

FlowsheetID	Product	Cyanide Consumption (kg/t)			
		50 µm	75 µm	125 µm	175 µm
Gravity	Gravity Con				
	Gravity Tail				
Option 1	Gravity Tail Leach	0.97	.084	0.81	0.88
	CND Float				
Option 2	Gravity Tail Float				
	Ro Conc Leach		0.84		
	CND Float				
	Ro Tail Leach	0.05	0.04	0.08	0.05
Total	**Option 1**	**0.97**	**0.84**	**0.81**	**0.88**
	Option 2 W. FT Leach		**0.87**		
	Option 2 W/O FT Leach		**0.87**		

Source: SGS, 2012

The data shows direct cyanidation of the gravity tailing considerably improves Au recovery with an increase of 8.5%. Alternatively, the recovery of Cu is shown to be much improved through rougher flotation.

Despite the significant improvement in Au recovery from direct leaching the capital and operating costs of such a process is expected to be significant due to the requirement of CND to allow subsequent recovery of Cu. Therefore, the Phase 2 Extension program evaluated extension of the flotation process to evaluate cyanidation of cleaner concentrate and cleaner tailing thus eliminating the need for the intermediate CND step between cyanidation and flotation.

The cyanide leaching testwork in the Phase 2 Extension was performed on three separate process streams. The first was the direct cyanidation of the gravity tailings resulting from two target grind P_{80}'s of 228 µm and 149 µm. The final Au extraction value for both tests was approximately 89% which indicates grind size has little effect in Au leaching of the samples.

The second was cyanidation of cleaner concentrate and cleaner tailing from MC-32 to MC-35. The results showed that cyanidation of the cleaner tailing was excellent with an Au extraction of 88.2%, while cyanidation of the cleaner concentrate was poor with an Au extraction of 59.0%.

The third cyanidation test was performed on the cleaner scavenger tailings from flotation tests MC-37 to MC-40. The result showed the final extraction of Au was about 81%.

Flotation tests MC-37 to MC-40 investigated the possibility of deporting a greater proportion of the Au to the cleaner tailing using a larger dosage of lime to increase the pH further. The results showed that the proportion of Au reporting to the cleaning circuit increased from 3% to approximately 5%.

13.7.4 Metallurgical Test Work Program 2012 to 2013

The 2011 MetTest Program at SGS focused on finding the most efficient processing alternative for recovery of Au and Cu from a composite of all mineralized material from the Toroparu Pit (aka Sample #8). More accurate resource and geologic models produced over the course of 2011/2012 identified that two geographically distinct types of Au mineralization occurred in the Toroparu and SE Pits, that were distinguishable based on sulphide and Cu content, and that these mineralized material types could be mined, stockpiled, and processed separately to improve processing efficiency and overall recovery.

A comprehensive metallurgical test program conducted at SGS Canada Inc., Lakefield, Ontario in 2012 tested processing methodologies and reagent consumptions for these two types of mineralized material to determine if improvements in metal recovery and reductions in reagent consumption would result from processing the mineralized material types separately, with the higher average Cu mineralized material being treated via flotation, and the “low” Cu mineralized material by cyanide leach processing.

The metallurgical test program conducted by SGS Lakefield in 2012 focused on two main mineralized material types which were classified as Au mineralized material with ACO and the Au mineralized material with LCO. The response of a Master Composite from each mineralized material zone and four Variability samples (A to D) to gravity separation, flotation, and cyanide leaching was examined in a detailed metallurgical test program.

13.7.5 Gold Mineralized Material with ACO

The testwork program on the ACO sample involved gravity separation, flotation, and cyanidation of the flotation and gravity separation products. Settling testwork was also conducted on a portion of the rougher tailing from the locked cycle test.

13.7.5.1 Gravity Separation (ACO)

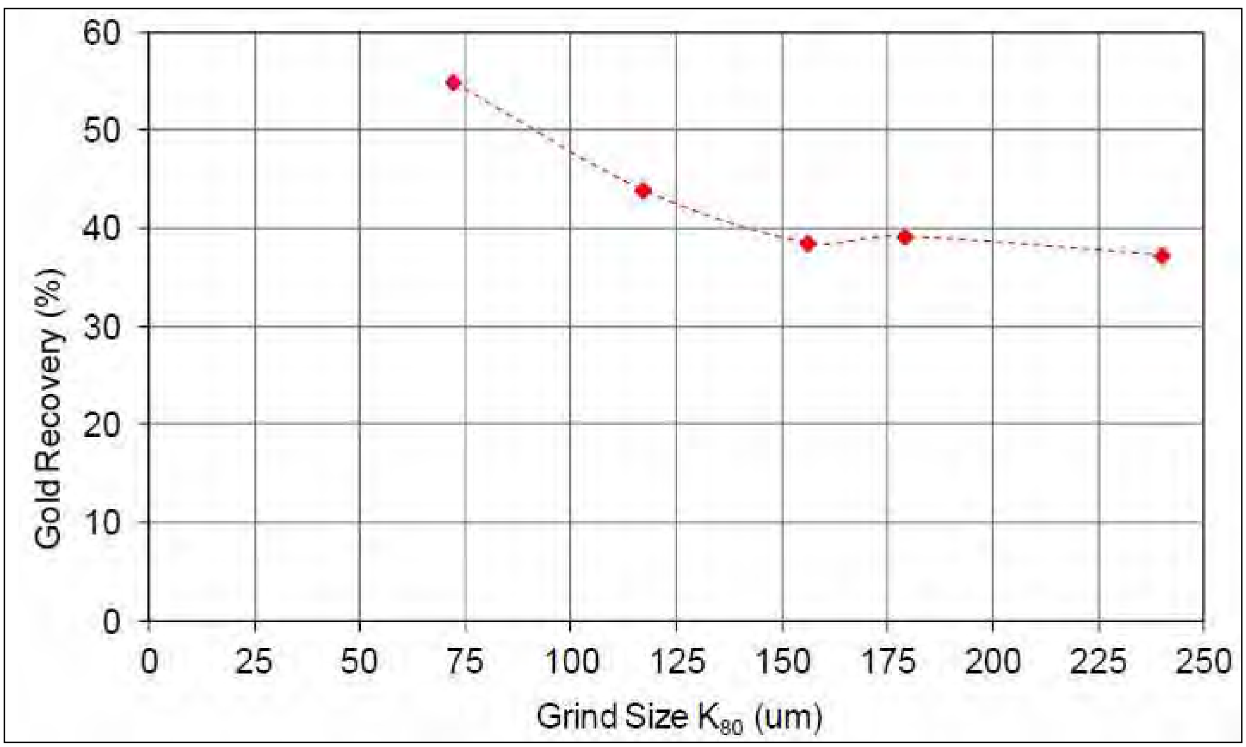
The ACO Composite was subjected to five gravity separation tests using a Knelson MD-3 Concentrator and Mozley C800 Laboratory Separator to examine the effect of grind size. Tests G-1 to G-5 were conducted over a range of grind size P_{80}'s of 75, 125, 175, 225 and 156 µm.

The Au recovery ranged from 37.3% for G-4 at P_{80} 240 µm to 54.9% for G-1 at P_{80} 72 µm, as detailed in Table 13-24. The Au recovery/grind size relationship is shown in Figure 13-8. The Au recovery increased with a reduction in feed P_{80} from P_{80} 156 µm to P_{80} 72 µm. The Au recovery from P_{80} 156 µm to P_{80} 240 µm was similar for all three tests.

Table 13-24: ACO Gravity Separation Test Results

Gravity Test No.	Feed Size P_{80}, µm	Feed Weight, kg	Product	Mass %	Assay Au, g/t	Distribution %
G-1	72	60	Mozley Concentrate	0.02	2,669	54.9
			Knelson/Mozely Tailing	99.98	0.47	45.1
			Head (Calculated)		1.05	100
G-2	117	60	Mozley Concentrate	0.02	2,724	43.9
			Knelson/Mozely Tailing	99.98	0.6	56.1
			Head (Calculated)		1.07	100
G-3	179	60	Mozley Concentrate	0.02	2,239	39.1
			Knelson/Mozely Tailing	99.98	0.63	60.9
			Head (Calculated)		1.03	100
G-4	240	60	Mozley Concentrate	0.02	2,569	37.3
			Knelson/Mozely Tailing	99.98	0.66	62.7
			Head (Calculated)		1.06	100
G-5	156	40	Mozley Concentrate	0.04	1,005	38.5
			Knelson/Mozely Tailing	99.96	0.63	61.5
			Head (Calculated)		1.02	100

Source: SGS, 2013



Source: SGS, January 2013

Figure 13-8: Grind size vs. gold recovery for ACO Composite

These tests were consistent with the 2011 tests results, which indicate gravity Au recoveries range from 30% to 50%. Based on the results of test G-5, a gravity recovery of 38% was selected for the ACO material for use in the economic evaluation.

13.7.5.2 Flotation Testwork (ACO)

Bulk Rougher Flotation (ACO)

Bulk rougher flotation tests were performed on the four gravity tailings produced from tests G-1 to G-4 as part of the evaluation of grind size. The following reagents were used in the rougher flotation tests.

- Aerofroth 208;
- PAX;
- Methyl Isobutyl Carbinol (MIBC); and
- Hydrated lime.

For each grind size, two duplicate flotation tests were conducted. The results of the rougher flotation tests are shown in Table 13-25: Rougher Flotation Results.

The recovery of Au ranged from 74.9% at 68 µm to 82.3% at P_{80} 116 µm. The recovery of Ag ranged from 76.8% at P_{80} 230 µm to 82.5% at P_{80} 116 µm. The recovery of Cu ranged from 91.1% at P_{80} 230 µm to 97.3% at P_{80} 69 µm. It appears that Au and Ag have the highest recovery at the 125 µm and 175 µm series of tests while Cu recovery significantly increases from the 225 µm series of tests to the 175 µm series of tests but increased slightly with finer grind sizes.

Table 13-25: Rougher Flotation Results

P_{80}	Product	Wt%	Assay, %, g/t				Distribution, %			
			Cu	Au	Ag	S	Cu	Au	Ag	S
69	Rougher Conc	5.5	3.75	7.40	33.6	4.40	97.3	80.7	79.7	96.3
	Rougher Tail	94.5	0.006	0.10	<0.5	0.01	2.7	19.3	20.3	3.7
	Head (Calc)	100	0.21	0.51	2.3	0.25	100	100	100	100
68	Rougher Conc	6.6	2.78	5.38	24.9	3.33	97.0	74.9	77.8	95.9
	Rougher Tail	93.4	0.006	0.13	<0.5	0.01	3.0	25.1	22.2	4.1
	Head (Calc)	100	0.19	0.47	2.1	0.23	100	100	100	100
116	Rougher Conc	6.0	3.30	8.96	33.5	4.00	96.8	82.2	81.0	92.7
	Rougher Tail	94.0	0.007	0.12	<0.5	0.02	3.2	17.8	19.0	7.3
	Head (Calc)	100	0.20	0.65	2.5	0.26	100	100	100	100
116	Rougher Conc	6.6	2.83	6.90	27.6	3.47	96.6	77.3	79.6	96.1
	Rougher Tail	93.4	0.007	0.14	<0.5	0.01	3.4	22.7	20.4	3.9
	Head (Calc)	100	0.19	0.59	2.3	0.24	100	100	100	100
174	Rougher Conc	5.7	3.52	9.96	37.8	4.53	95.1	79.6	82.0	90.1
	Rougher Tail	94.3	0.011	0.15	<0.5	0.03	4.9	20.4	18.0	9.9
	Head (Calc)	100	0.21	0.71	2.62	0.29	100	100	100	100
175	Rougher Conc	6.7	2.83	8.11	27.2	3.79	95.3	80.1	79.5	96.4
	Rougher Tail	93.3	0.010	0.14	<0.5	0.01	4.7	19.9	20.5	3.6
	Head (Calc)	100	0.20	0.67	2.28	0.26	100	100	100	100
233	Rougher Conc	5.7	3.41	8.92	31.2	4.00	93.6	75.0	78.9	85.7
	Rougher Tail	94.3	0.014	0.18	<0.5	0.04	6.4	25.0	21.1	14.3
	Head (Calc)	100	0.21	0.67	2.2	0.26	100	100	100	100
230	Rougher Conc	5.6	2.95	10.3	28.0	3.85	91.1	77.2	76.8	91.9
	Rougher Tail	94.4	0.017	0.18	<0.5	0.02	8.9	22.8	23.2	8.1
	Head (Calc)	100	0.18	0.74	2.0	0.23	100	100	100	100

Source: SGS, 2013

The combined gravity and flotation recoveries of Au and Ag are tabulated in Table 13-26.

Table 13-26: ACO Combined Gravity and Flotation and Recovery of Au and Ag

Grind Size Campaign	Gravity Recovery, %		Flotation Recovery, % *		Comb. Recovery, %	
	Au	Ag	Au	Ag	Au	Ag
75 µm	54.9	12.7	77.8	78.8	90.0	81.5
125 µm	43.9	8.3	80.6	81.0	89.1	82.6
175 µm	39.1	10.5	80.5	81.1	88.1	83.1
225 µm	37.3	9.6	76.1	77.9	85.0	80.0

Source: SGS, 2013. *Average of all tests.

Since the performance of the 125 and 175 µm grind sizes were similar in Au and Ag recoveries and superior for Cu recovery, a grind size P_{80} of 150 µm was selected for subsequent flotation testwork.

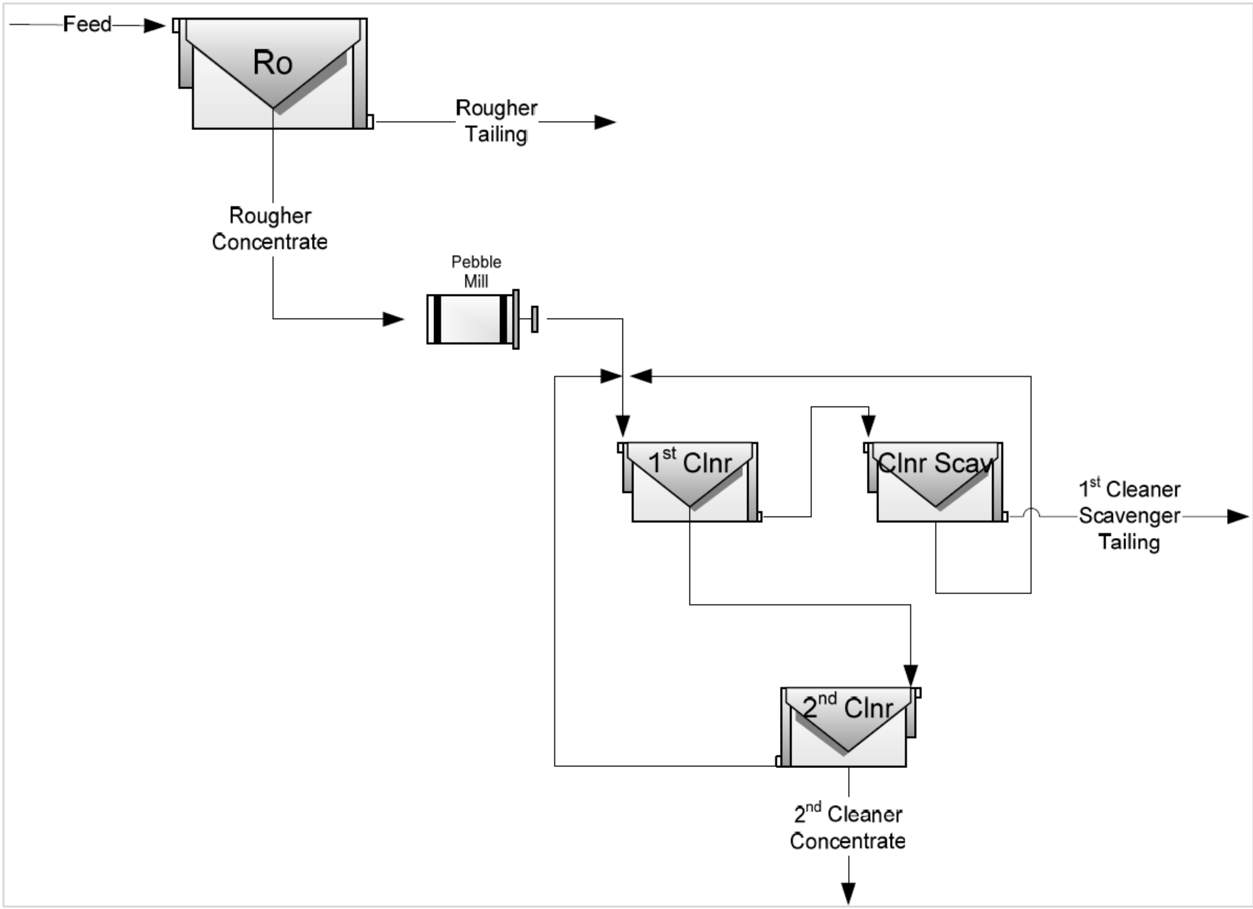
13.7.5.3 Open Circuit Cleaning Flotation (ACO)

Two open circuit flotation tests (ACO-01 and ACO-02) were performed on the gravity tailings produced in test G-5 on the composite ACO-in order to confirm conditions for the locked cycle test (LCT) to produce a Cu concentrate at a target grind of P_{80} 150 µm. The same reagents used in the bulk rougher flotation tests were also used in these tests with the addition of Carboxy Methyl Cellulose (CMC).

The flotation tests consisted of four rougher stages, a pebble mill regrind of the combined rougher concentrate, followed by two cleaner stages and a cleaner scavenger. Two tests examined the effect of regrind time and CMC dosage. Results show that the regrind has little to no effect on the performance of the cleaner flotation and CMC dosage is critical. Due to the higher dosage of CMC, test ACO-02 produced a higher-grade concentrate of 25.3% Cu, 88.5 g/t Au, 224 g/t Ag and 26.1% S at recoveries of 73.9%, 59.2%, 55.1% and 62.5%, respectively.

13.7.5.4 Locked Cycle Testing (ACO)

The ACO Composite was subjected to a six-cycle LCT. The test was performed using 4 kg charges of the ACO Composite gravity tailings ground to 156 µm (from test G-5). The LCT flowsheet is displayed in Figure 13-9. The reagents used included PAX in conjunction with the promoter Aerofloat 208. In the two cleaning and cleaner scavenger stages, CMC was used as a non- sulphide gangue depressant. Lime was used to modify the pH and MIBC was used as a frother and added on an as required basis.



Source: SGS, January 2013

Figure 13-9: LCT flowsheet for ACO composite

The results from the LCT are shown in Table 13-27. The Cu concentrate obtained from the last three stages (D to F) produced a concentrate containing 21.0% Cu, 56.0 g/t Au, 180 g/t Ag and 20.9% S at recoveries of 91.3%, 67.2%, 65.0% and 78.4% respectively. The first cleaner scavenger tailing contained 0.091% Cu, 0.77 g/t Au and 2.1 g/t Ag and constituted 10.6% of the total mass containing 12.6% of the Au and 10.2% of the Ag. A bottle roll test, CN-15 was performed on this product.

Table 13-27: LCT Results

Product	Wt		Assays, %, g/t				% Distribution			
	g	%	Cu	Au	Ag	S	Cu	Au	Ag	S
Copper Con	190	0.8	21.0	56.0	180	20.9	91.3	67.2	65.0	78.4
1st Cl Sc Tailings	2,599.6	10.6	0.091	0.77	2.1	0.34	5.4	12.6	10.2	17.3
Rougher Tailings	21,718.4	88.6	0.007	0.15	0.6	0.01	3.3	20.2	24.8	4.3
Head (Calculated)	24,508.0	100	0.18	0.65	2.1	0.21	100	100	100	100

Source: SGS, 2013

An examination of the stage-by-stage results indicated that the Cu and S grades dropped abruptly for the last two stages of the test (E and F), due to a build up of non sulphide gangue.

Au and Ag recoveries from the gravity separation test (G-5) combined with LCT at 150 µm was calculated to be 79.1% Au and 69.2% Ag. The results can be found in Table 13-28.

Table 13-28: ACO Combined Results from Gravity Separation and LCT Tests

Grind Size Campaign	Gravity Recovery, %		Cleaner Flotation, %		Comb. Recovery, %	
	Au	Ag	Au	Ag	Au	Ag
150 µm	38.5	11.9	67.2	65.0	79.1	69.2

Source: SGS, 2013

13.7.5.5 Cyanidation Testwork (ACO)

The response of the gravity concentrate stream, gravity tailings stream, and cleaner scavenger tailings stream to cyanide leaching was examined in a series of tests.

Gravity Concentrate Intensive Cyanidation (ACO)

The Mozley concentrate obtained from gravity tests G-1 to G-4 were submitted for intensive cyanidation. The tests were performed at 5% Solids with 20 g/L of NaCN for 24 hours. The results are shown in Table 13-29.

Table 13-29: ACO Intensive Cyanidation Results

CN Test No.	Gravity Test No.	Feed Size µm	Reag. Consumption kg/t of Feed	Au Ext %	Residue Au, g/t	Head Au g/t	Ag Ext %	Residue Ag, g/t	Head Ag g/t	Cu Soln
			NaCN	24 h		Calc	24 h		Calc	mg/L
CN-1	G-1	72	206	99	33.7	2,669	97	48.9	1,445	1,930
CN-2	G-2	117	182	99	28.2	2,724	97	35.7	1,168	1,580
CN-3	G-3	179	186	99	17.4	2,239	97	35.6	1,403	1,700
CN-4	G-4	240	289	99	18.3	2,569	97	36.8	1,393	1,810

Source: SGS, 2013

Au extraction was 99% for all cases and Ag extraction was 97%. Results show that the feed size has no effect on leaching recovery.

13.7.5.6 Gravity Tailing Bulk Cyanidation (ACO)

The gravity tailings of gravity test G-1 to G-4 at four P_{80} sizes (75, 125, 175, and 225 µm) were submitted for bulk cyanidation tests. The tests were performed at 40% solids with 0.5 g/L of NaCN as carbon in pulp (CIP) tests in which the carbon was added after 48 hours of leaching. Carbon contact time was about six hours.

The Au and Ag recoveries are shown in Table 13-30. The final extractions at 54 hours ranged from 71% for CN-12 at P_{80} 240 µm to 87% for CN-5 at P_{80} 72 µm and CN-10 at P_{80} 179 µm. Overall, the Au extraction occurred in a narrow range and showed a general trend of an increase in Au extraction with decreasing feed size.

The final Ag extractions at 54 hours were distributed in a very narrow range from 72% for CN-12 at P_{80} 240 µm to 77% for CN-5 at P_{80} 72 µm and CN-7 at P_{80} 117 µm.

The final Cu extractions for all the leaches ranged from 16% for CN-9 at P_{80} 166 µm to 20% for CN-7 at P_{80} 117 µm. The Cu extraction for all tests was low and does not appear to be influenced by varying feed size.

Table 13-30: ACO Gravity Tailing Cyanidation Gold and Silver Results

CN Test No.	Gravity Test No.	Feed Size	Reag. Consumption kg/t of CN Feed	Au Ext %	Residue Au, g/t	Head Au, g/t	Ag Ext %	Residue Ag, g/t	Head Ag, g/t
		P_{80}, µm	NaCN	54 h		Calc	54 h		Calc
CN-5	G-1	72	1.14	87	0.06	0.48	77	<0.5	2.2
CN-6		68	1.16	85	0.07	0.44	75	<0.5	2.0
CN-7	G-2	117	1.49	84	0.10	0.60	77	<0.5	2.2
CN-8		113	1.10	82	0.10	0.57	76	<0.5	2.1
CN-9	G-3	166	1.07	79	0.12	0.55	74	<0.5	1.9
CN-10		179	1.28	87	0.08	0.58	74	<0.5	1.9
CN-11	G-4	240	1.58	72	0.19	0.66	78	<0.5	2.0
CN-12		240	1.09	71	0.17	0.57	75	<0.5	1.8

Source: SGS, 2013

The combined extraction of Au and Ag from the gravity separation stage and cyanide leaching ranged from 82% for CN-11 and CN-12 (both P_{80} 240 µm) to 94% for CN-5 at P_{80} 72 µm for Au and from 75% for CN-12 at P_{80} 240 µm to 80% for CN-5 at P_{80} 75 µm for Ag. The results are shown in Table 13-31

Table 13-31: ACO Combined Results from Gravity Separation and Gravity Tailing Cyanidation Tests

CN Test No.	Gravity Test No	Feed Size P_{80}, µm	Gravity Concentrate Recovery, %		Gravity Tail CN Leach Recovery, %		Comb. Recovery, %	
			Au	Ag	Au	Ag	Au	Ag
CN-5	G-1	72	54.9	12.7	87	77	94	80
CN-6		68	54.9	12.7	85	75	93	78
CN-7CN-8	G-2	117	43.9	8.3	84	77	91	79
		113	43.9	8.3	82	76	90	78
CN-9	G-3	166	39.1	10.5	79	74	87	76
CN-10		179	39.1	10.5	87	74	92	77
CN-11	G-4	240	37.3	9.6	72	78	82	78
CN-12		240	37.3	9.6	71	75	82	75

Source: SGS, 2013

LCT Cleaner Tailing Cyanidation (ACO)

One cyanidation test was performed on the cleaner tailings obtained from the LCT conducted with the ACO Composite. The amount of Cu and Au reporting to the 1st Cleaner Scavenger Tailing in this test was 5.4% and 12.6% respectively. The result of the cyanidation test is shown in Table 13-32. The extraction of Au and Ag was 72% and 64% respectively and was accompanied by a Cu extraction of 34%.

Table 13-32: ACO Cleaner Tailing Cyanidation Results

CN Test No.	Reag. Consumption kg/t of CN Feed	Head Au, g/t	Residue Au, g/t	Au Ext %	Head Ag, g/t	Residue Ag, g/t	Ag Ext %
	NaCN	Calc		54 h	Calc		54 h
CN-15	0.99	0.84	0.24	72	2.8	1.0	64

Source: SGS, 2013

The overall recovery of the gravity separation coupled with cleaner flotation and cyanide leaching of the cleaner tailing is summarized in Table 13-33.

Table 13-33: ACO Gravity, Cleaner Flotation, and Cleaner Tail Leach Summary

Grind Size Campaign	Gravity Recovery, %		Cleaner Con Recovery, %		Cleaner Tail Recovery, %		Cleaner TailCN Leach		Combined Recovery, %	
	Au	Ag	Au	Ag	Au	Ag	Au	Ag	Au	Ag
150 µm	**38.5**	**11.9**	**67.2**	**65.0**	**12.6**	**10.2**	**72.2**	**64.2**	**85.4**	**74.9**

Source: SGS, 2013

From comparison of the results of the ACO test program, cyanide leaching of the gravity separation tailing offers higher Au and Ag recovery than rougher flotation and also cleaner flotation combined with leaching of the cleaner tailing. However, the cyanide consumption and Cu extraction from the cleaner flotation processing route is 0.11 kg/t and 1.8% Cu respectively. This is considerably lower than the gravity tailing leaching route that resulted in cyanide consumption and Cu extraction of 1.24 kg/t and 18.0% Cu respectively.

13.7.6 Gold Mineralized Material with LCO

The LCO metallurgical testwork program involved testing of a Master Composite and the four Variability Composites (A to D) to evaluate response to gravity separation, rougher flotation, and cyanidation of gravity separation concentrate. Evaluation of the effect of grind size on metallurgical performance was limited during this testwork program and included three scoping rougher flotation tests at P_{80} grind sizes of 125, 175 and 225 µm to confirm the findings from the ACO test program.

13.7.6.1 Gravity Separation (LCO)

The LCO Master and Variability Composites (A to D) were subjected to total of 14 gravity separation tests (G-6 to G-19) using a Knelson concentrator and Mozley C800, examining the effect of grind size. No lime was added during the grind or gravity separation stages. The effect of five varying P_{80} grind sizes (75, 125, 150, 175, and 225 µm) was examined on the Master Composite while the Variability Composites (A to D) were examined for two P_{80} grind sizes (75 and 150 µm).

The results of these tests are shown in Table 13-34.

Table 13-34: LCO Gravity Separation Test Results

Gravity Test No.	Composite	Feed Size P_{80}, µm	Feed Weight, kg	Product	Mass %	Assay Au, g/t	Distribution %
G-6	Master	76	30	Mozley Concentrate	0.06	746	58.6
				Knelson/Mozely Tailing	99.94	0.3	41.4
				Head (Calculated)		0.71	100
G-7	A	73	10	Mozley Concentrate	0.05	543	39.2
				Knelson/Mozely Tailing	99.95	0.38	60.8
				Head (Calculated)		0.62	100
G-8	B	75	10	Mozley Concentrate	0.09	343	47.7
				Knelson/Mozely Tailing	99.91	0.33	52.3
				Head (Calculated)		0.63	100
G-9	C	68	10	Mozley Concentrate	0.12	743	60.8
				Knelson/Mozely Tailing	99.88	0.57	39.2
				Head (Calculated)		1.45	100
G-10	D	78	10	Mozley Concentrate	0.12	285	37.7
				Knelson/Mozely Tailing	99.88	0.58	62.3
				Head (Calculated)		0.93	100
G-11	Master	252	4	Mozley Concentrate	0.05	321	28.9
				Knelson/Mozely Tailing	99.95	0.43	71.1
				Head (Calculated)		0.6	100
G-12	Master	179	4	Mozley Concentrate	0.07	421	42.6
				Knelson/Mozely Tailing	99.93	0.37	57.4
				Head (Calculated)		0.64	100
G-13	Master	131	4	Mozley Concentrate	0.06	400	39.9
				Knelson/Mozely Tailing	99.94	0.35	60.1
				Head (Calculated)		0.57	100
G-14	Master	150	30	Mozley Concentrate	0.08	469	50
				Knelson/Mozely Tailing	99.92	0.36	50

Gravity Test No.	Composite	Feed Size P_{80}, µm	Feed Weight, kg	Product	Mass %	Assay Au, g/t	Distribution %
				Head (Calculated)		0.72	100
G-15	A	153	10	Mozley Concentrate	0.06	472	42.5
				Knelson/Mozely Tailing	99.94	0.37	57.5
				Head (Calculated)		0.64	100
G-16	B	142	10	Mozley Concentrate	0.08	302	38.8
				Knelson/Mozely Tailing	99.92	0.36	61.2
				Head (Calculated)		0.59	100
G-17	C	135	10	Mozley Concentrate	0.05	1,696	57.9
				Knelson/Mozely Tailing	99.95	0.67	42.1
				Head (Calculated)		1.59	100
G-18	D	152	10	Mozley Concentrate	0.12	423	46.1
				Knelson/Mozely Tailing	99.88	0.61	53.9
				Head (Calculated)		1.14	100
G-19	Master	150	10	Mozley Concentrate	0.1	390	50.2
				Knelson/Mozely Tailing	99.9	0.37	49.8
				Head (Calculated)		0.74	100
G-20	A	150	10	Mozley Concentrate	0.07	342	48.1
				Knelson/Mozely Tailing	99.93	0.27	51.9
				Head (Calculated)		0.52	100

Source: SGS, 2013

The Au recovery for the Master Composite ranged from 28.9% for P_{80} grind size 225 µm to 58.9% for P_{80} grind size 75 µm.

The Au recovery for Variability Composites at P_{80} grind size 75 µm were 39.2%, 47.7%, 60.8% and 37.7% for Composites A (G-7), B (G-8), C (G-9) and D (G-10) respectively. At a P_{80} grind size of 150 µm, Au recoveries of 42.5%, 48.1%, 38.8%, 57.9% and 46.1% were achieved with Composites A (G-15 and G-20), B (G-16), C (G-17) and D (G-18) respectively.

13.7.6.2 Flotation Testwork (LCO)

Scoping Rougher Flotation (LCO)

Three scoping rougher flotation tests were performed on the three gravity tailings produced at three different P_{80} grind sizes (125,175, and 225 µm) to observe the effect of grind size on rougher flotation performance. The following reagents were used in the rougher flotation tests:

- Aerofloat 208;
- PAX; and
- MIBC.

The recovery of Cu and Au increased as a function of finer grind size, while the recovery of Ag did notdisplay this trend. The Cu recovery increased from 93.6% at 252 µm to 97.2% at P_{80} 125 µm. Similarly,the Au recovery increased from 79.7% at 252 µm to 84.3% at P_{80} 125 µm. In order to compare resultsof these tests to the testing of the ACO sample, a P_{80} grind size of 150 µm was selected for the subsequent bulk rougher flotation tests.

13.7.6.3 Bulk Rougher Flotation (LCO)

A total of eight bulk rougher flotation tests were performed on the gravity tailing produced from the Master and Variability Composites, all at the P_{80} grind size of 150 µm. The objective of the tests was to produce concentrate for the subsequent cyanidation testwork.

The results show that the recovery of Cu ranged from 97.0% for the Master Composite to 92.4% for Variability Composite A. The recovery of Au shows significant variation between samples from as lowas 65.7% and 68.1% for Variability Composite A and C respectively to as high as 80.2% and 76.3% for Variability mineralized material B and D, respectively. The average of three tests of the Master Composite showed an Au recovery of 75%.

The overall combined gravity and flotation Au recovery from the Master and Variability Composites is shown in Table 13-34.

Table 13-35: Combined Gravity and Flotation Au Recovery for the LCO Composites

Composite	Au Gravity Recovery, %	Au Flotation Recovery, % *	Au Comb. Recovery, %
Master	67.5	75.1	91.9
Ore A	42.5	69.0	82.2
Ore B	39.0	80.8	88.3
Ore C	57.9	68.1	86.6
Ore D	46.2	76.3	87.3

Source: SGS, 2013 * Average of all applicable tests

13.7.6.4 Cyanidation Testwork (LCO)

Gravity Concentrate Intensive Cyanidation (LCO)

The Mozley concentrate obtained from gravity tests on Master and Variability Composites of P_{80} grind sizes of 75 and 150 µm were submitted for intensive cyanidation under the same test protocol used for the intensive cyanidation conducted on ACO Composite. The tests were performed at 5% solids with 20 g/L of NaCN for 24 hours. The results are shown in Table 13-36.

Table 13-36: LCO Intensive Cyanidation Results

CN Test No.	Gravity Test No.	Feed Size µm	Reag. Consumption kg/t of CN Feed	Head Au, g/t	Residue Au, g/t	AuExt %	Head Ag, g/t	Residue Ag, g/t	AgExt %
			NaCN	Calc		24 h	Calc		24 h
CN-25	Master	150	73	469	11.0	98	152	18.1	88
CN-38	Master	150	119	390	18.5	95	131	22.7	83
CN-20	Master	76	106	746	9.00	99	216	21.6	90
CN-26	A	153	167	472	6.09	99	150	25.5	83
CN-39	A	150	157	342	16.2	95	122	29.1	76
CN-21	A	73	192	543	5.31	99	139	22.9	84
CN-27	B	142	135	302	8.52	97	166	15.1	91
CN-22	B	75	126	343	9.29	97	156	18.7	88
CN-28	C	135	213	1,696	8.95	99.5	71	18.8	92
CN-23	C	68	115	743	2.99	99.6	99	10.0	90
CN-29	D	152	102	423	12.7	97	71	12.0	83
CN-24	D	78	111	285	0.12	99.96	62	21.4	66

Source: SGS, 2013

The Au extraction from all the tests were quite high, ranging from 95% to 100% and the extraction of Ag ranged from 83% to 92% except for CN-39 and CN-24 that were 76% and 66% respectively. The extent of leaching does not appear to be affected by the feed size.

13.7.6.5 Gravity Tailing Bulk Cyanidation

The gravity tailings resulting from the gravity separation test on LCO Master and Variability Composites were submitted for bulk cyanidation tests. The LCO Master Composite was leached at P_{80} 75 µm and P_{80} 150 µm, while the LCO Variability Composites (A to D) were leached at P_{80} 75 µm. The bulk leaches were performed under the CIP protocol.

The resulting Au and Ag recoveries are shown in Table 13-37. Table 13-37The final extraction at 54 hours for leaching at P_{80} 75 µm ranged from 87% for CN-14 to 95% for CN-16 and 84% for CN-37 conducted atP_{80} 150 µm. Extraction of Ag ranged from 63% for CN-13 to 29% for CN-19. Extraction of Cu ranged from 6% for CN-19 to 9% for CN-13 and CN-17 for the P_{80} 75 µm leaches, and for CN-37 (150 µm) the extraction was 9%.

Table 13-37: LCO Gravity Tailing Cyanidation Gold and Silver Results

CN Test No.	Gravity Test No.	Feed Size P_{80}, µm	Reag. Consumption kg/t of CN Feed	Head Au, g/t	Residue Au, g/t	AuExt %	Head Ag, g/t	Residue Ag, g/t	Ag Ext %
			NaCN	Calc		54 h	Calc		54 h
CN-37	Master	150	0.44	0.37	0.06	84	1.1	<0.5	56
CN-13	Master	76	0.39	0.30	0.03	90	1.4	<0.5	63
CN-14	Master	75	0.53	0.31	0.04	87	1.1	<0.5	54
CN-16	A	73	0.61	0.38	0.02	95	0.8	<0.5	36
CN-17	B	75	0.67	0.33	0.03	91	1.0	<0.5	51
CN-18	C	68	0.63	0.57	0.05	92	0.9	<0.5	42
CN-19	D	78	0.51	0.58	0.07	88	0.7	<0.5	29

Source: SGS, 2013

The combined extraction of Au and Ag from the gravity separation stage and cyanide leaching rangedfrom 92% for CN-19 at 78 µm and CN-37 at P_{80} 150 µm to 97% for CN-16 and CN-18 both at P_{80} 72 µm for Au and from 36% for CN-19 at P_{80} 78 µm to 66% for CN-13 at P_{80} 76 µm for Ag. The results are shown in Table 13-38. bookmark263

Table 13-38: LCO Combined Results from Gravity Separation and Gravity Tailing Cyanidation Tests

CN Test No.	Ore Type	Feed Size P_{80}, µm	Gravity Recovery, %		Gravity Tailing CN Leach Recovery, %		Comb. Recovery, %	
			Au	Ag	Au	Ag	Au	Ag
CN-37	Master	150	50	11.4	84	56	92	61
CN-13	Master	76	58.6	7.9	90	63	96	66
CN-14	Master	75	58.6	7.9	87	54	95	57
CN-16	A	73	39.2	7.3	95	36	97	41
CN-17	B	75	47.7	12	91	51	95	57
CN-18	C	68	60.8	11.6	92	42	97	49
CN-19	D	78	37.7	9.8	88	29	92	36

Source: SGS, 2013

These tests were used to assign a total recovery of 95.9% Au for the LCO Composite for the economic evaluation. This is based on the average of tests CN-13, CN-14, and CN-16. The inclusion of test CN-16 into this average was based on it including the lower gravity recovery of 39.2% and because itrepresented the lowest head grade tested.

13.7.6.6 Rougher Concentrate Cyanidation (LCO)

The rougher concentrate obtained from the flotation tests of LCO Master and Variability Composites were submitted for CIP cyanidation tests. A total of nine rougher concentrate cyanidation tests were performed including four on the Master Composite and five on the Variability Composites. The Au and Ag extraction results are shown in Table 13-39. The extraction of Au ranged from 89% to 95% for tests CN-35, CN-32, and CN-36R respectively. The Ag extraction ranged from 60% to 82% for test CN-36 and CN-34. The 54-hour extraction results are the sum of solution and carbon assays.

Table 13-39: LCO Rougher Concentrate Leaching Gold and Silver Results

CN Test No.	Gravity Test No.	Feed Size P_{80}, µm	Reag. Consumption kg/t of CN Feed	HeadAu, g/t	Residue Au, g/t	Au Ext %	HeadAg, g/t	Residue Ag, g/t	Ag Ext %
			NaCN	Calc		54 h	Calc		54 h
CN-30	Master	22	9.18	3.19	0.27	92	4.9	1.3	74
CN-31	Master	23	9.96	3.24	0.26	92	5.0	1.4	72
CN-36	Master	-	23.3	5.34	0.40	93	5.2	2.1	60
CN-36R	Master	23	9.86	4.63	0.24	95	6.2	1.2	81
CN-32	A	22	10.2	4.36	0.24	95	4.7	1.6	66
CN-32R	A	23	7.35	2.79	0.25	91	4.2	1.6	62
CN-33	B	21	13.1	4.07	0.35	91	7.4	1.5	80
CN-34	C	17	11.0	7.40	0.44	94	5.7	1.0	82
CN-35	D	23	7.93	9.14	1.00	89	3.4	1.0	70

Source: SGS, 2013

The Cu extraction for all the leaches did not exceed 14% at 54 hours. The iron extractions were very low and did not exceed 1.5%; however, there was still a significant amount of iron present in solution which resulted in high cyanide consumptions.

The combined recovery of gravity separation, rougher flotation, and cyanidation of the rougher concentrate for Au and Ag are shown in Table 13-40. The combined recovery for Au ranged from 79% (CN-32R) to 85% (CN-31) and for Ag ranged from 30% to 50% for CN-35 and CN-33 tests, respectively.

Table 13-40: LCO Combined Results from Gravity Separation, Rougher Flotation and Rougher Concentrate Leaching

CN Test No.	Ore Type	Recovery, %							
		Gravity		Gravity Tail Flotation		Leach		Combined *	
		Au	Ag	Au	Ag	Au	Ag	Au	Ag
CN-30	Master	50	11.4	75.7	50	92	74	84.8	44.2
CN-31	Master	50	11.4	77	49.2	92	72	85.4	42.8
CN-36	Master	50.2	12.2	72.7	48.2	93	60	83.9	37.6
CN-36R	Master	50.2	12.2	72.7	48.2	95	81	84.6	46.5
CN-32	A	42.5	8.8	72.3	41.2	95	66	82.0	33.6
CN-32R	A	48.1	11	65.7	35.8	91	62	79.1	30.8
CN-33	B	38.8	11.2	80.8	55.1	91	80	83.8	50.3
CN-34	C	57.9	4.1	68.1	45.6	94	82	84.8	40.0
CN-35	D	46.1	11.2	76.3	29.8	89	70	82.7	29.7

Source: SGS, 2013, * Leach of rougher float concentrate

13.7.7 Comparison of Cyanide Leaching of Gravity Tailing and Flotation Products for ACO and LCO

The main objective of the ACO and LCO testwork program was to compare the performance of direct cyanide leaching of the gravity tailing and cyanide leaching of a flotation product. The flotation product of interest was flotation cleaner scavenger tailings for ACO program while it was a rougher concentrate for the LCO program.

Table 13-41 through Table 13-43 show comparisons of the overall Au recovery for three different combinations for ACO testwork program.

Table 13-41: Overall Gold Recovery for ACO Composite, Gravity and Gravity Tailing Cyanide Leaching

Grind Size Campaign	Au Recovery, %					
	Gravity		Gravity TailCN Leach *		Combined	
	Au	Ag	Au	Ag	Au	Ag
75 µm	54.9	12.7	86.3	76.0	93.8	79.0
125 µm	43.9	8.3	83.3	76.4	90.6	78.3
175 µm	39.1	10.5	82.8	73.8	89.6	76.5
225 µm	37.3	9.6	71.6	73.8	82.2	76.3

Source: SGS, 2013, * Average of all tests

Table 13-42: Overall Gold Recovery for ACO Composite, Gravity and Gravity Tailing Rougher Flotation

Grind Size Campaign	Au Recovery %					
	Gravity Recovery, %		Flotation Recovery, % *		Comb. Recovery, %	
	Au	Ag	Au	Ag	Au	Ag
75 µm	54.9	12.7	77.8	78.8	90.0	81.5
125 µm	43.9	8.3	80.6	81.0	89.1	82.6
175 µm	39.1	10.5	80.5	81.1	88.1	83.1
225 µm	37.3	9.6	76.1	77.9	85.0	80.0

Source: SGS, 2013, * Average of all tests

Table 13-43: Overall Gold Recovery for ACO Composite, Gravity, Gravity Tailing Cleaner Flotation and Cleaner Tail Cyanide Leaching

Grind Size Campaign	Au Recovery %									
	Gravity		Cleaner Con		Cleaner tail		Cleaner Tail CN Leach		Combined	
	Au	Ag	Au	Ag	Au	Ag	Au	Ag	Au	Ag
150 µm	38.5	11.9	67.2	65.0	12.6	10.2	72.2	64.2	85.4	74.9

Source: SGS, 2013

The results show that the gravity separation and leaching of the gravity separation tailing for all grind sizes offers higher overall Au recovery.

Similarly, Table 13-44 and Table 13-45 show comparisons of the overall Au recovery for two different combinations for LCO testwork program.

Table 13-44: Overall Gold Recovery for LCO Composites, Gravity and Gravity Tailing Cyanide Leaching

Sample	Au Recovery, %					
	Gravity		Gravity Tail CN Leach		Combined	
	Au	Ag	Au	Ag	Au	Ag
Master *	58.6	7.9	88.4	58.4	95.2	61.7
Ore A	39.2	7.3	94.7	36.2	96.8	40.7
Ore B	47.7	12	90.9	51.3	95.2	56.9
Ore C	60.8	11.6	92.1	42.2	96.9	48.7
Ore D	37.7	9.8	87.8	29.0	92.4	36.0

Source: SGS, 2013, * Average of all tests where applicable

Table 13-45: Overall Gold Recovery for LCO Composites, Gravity and Gravity Tailing Rougher Flotation and Rougher Concentrate Cyanide Leaching

Sample	Au Recovery, %							
	Gravity		Gravity Tail Flotation		Ro Conc Leach		Combined	
	Au	Ag	Au	Ag	Au	Ag	Au	Ag
Master *	50.1	11.8	75.1	49.1	92.7	71.4	84.9	42.7
Ore A*	45.3	9.9	69.0	38.5	92.9	64.1	80.4	32.2
Ore B	38.8	11.2	80.8	55.1	91.4	79.8	84.0	50.3
Ore C	57.9	4.1	68.1	45.6	94.1	82.5	84.9	40.1
Ore D	46.1	11.2	76.3	29.8	89.1	70.3	82.8	29.8

Source: SGS, 2013, * Average of all tests where applicable

Similar to the ACO program, the overall Au recovery for the LCO samples is also higher for the gravity separation and leaching of the gravity separation tailing option. However, the cyanide consumption and capital cost of the direct leaching are the essential factors in choosing the optimized method for Au recovery.

The comparison of cyanide consumption associated with direct gravity separation tailing leaching and flotation product leaching is given in Table 13-46 and Table 13-47 for the ACO program and in Table 13-48 and Table 13-49 for the LCO program. The results show that the cyanide consumption and Cu extraction for the case of cyanide leaching of the cleaner tailing for the ACO Composite is considerably lower than the direct cyanide leaching of the gravity separation tailing case. Conversely, in the case of the LCO composites, the cyanide consumption and Cu extraction of the rougher concentrate is higher than the direct gravity tailing case.

Table 13-46: Comparison of Copper Extraction and Cyanide Consumption for ACO

Grind Size Campaign	Gravity Tail CN Leach	
	Cu Extraction, % *	CN Consumption, kg/t *
75 µm	18.8	1.15
125 µm	18.4	1.30
175 µm	17.5	1.17
225 µm	17.2	1.33
Average	18.0	1.24

Source: SGS, 2013, * Average of all tests where applicable

Table 13-47: Copper Extraction and Cyanide Consumption for Cleaner Tails Leaching of ACO

Product	Cleaner Tail CN Leach		Corrected for 10.6% Wt, 5.4% Cu Dist.	
	Cu Extraction, %	CN Consumption, kg/t	Cu Extraction, %	CN Consumption, kg/t
Cleaner Tail	33.9	0.99	1.8	0.11

Source: SGS, 2013

Table 13-48: Copper Extraction and Cyanide Consumption for Gravity Tails Leaching of LCO

Grind Size Campaign	Gravity Tail CN Leach	
	Cu Extraction, %	CN Consumption, kg/t
Master *	8.1	0.46
Ore A	7.4	0.61
Ore B	9.1	0.67
Ore C	8.3	0.63
Ore D	6.3	0.51
Average	7.8	0.57

Source: SGS, 2013, * Average of all tests

Table 13-49: Copper Extraction and Cyanide Consumption for Rougher Concentrate Leaching of LCO

Product	Rougher Concentrate CN Leach		Corrected for % Wt and Cu Dist.	
	Cu Extraction, %	CN Consumption, kg/t	Cu Extraction, %	CN Consumption, kg/t
Master *	10.8	13.1	10.5	1.09
Ore A*	9.6	8.77	8.9	0.72
Ore B	12.7	13.1	12.2	1.11
Ore C	14.5	11.0	13.6	0.75
Ore D	6.6	7.93	6.4	0.52

Source: SGS, 2013, * Average of all tests

13.8 Saprolite Test Work Program 2012-2013

The metallurgical test program conducted by Inspectorate in 2012 and 2013 focused on the extraction and recovery of Au from saprolite. The testwork included gravity separation, cyanidation, flotation, and cyanide detoxification.

It is noted that while the saprolite composite was divided into coarse and fine composites, the coarse fraction top size is 10 mesh, which should pass through the screening portion of the saprolite circuit in the proposed flowsheet. As such, the separated “oversize” material (consisting of roots, rocks, or tramp metal), does not have an impact on overall saprolite Au recovery since the screen oversize is defined as 4 inches or greater.

13.8.1 Gravity Separation Testwork

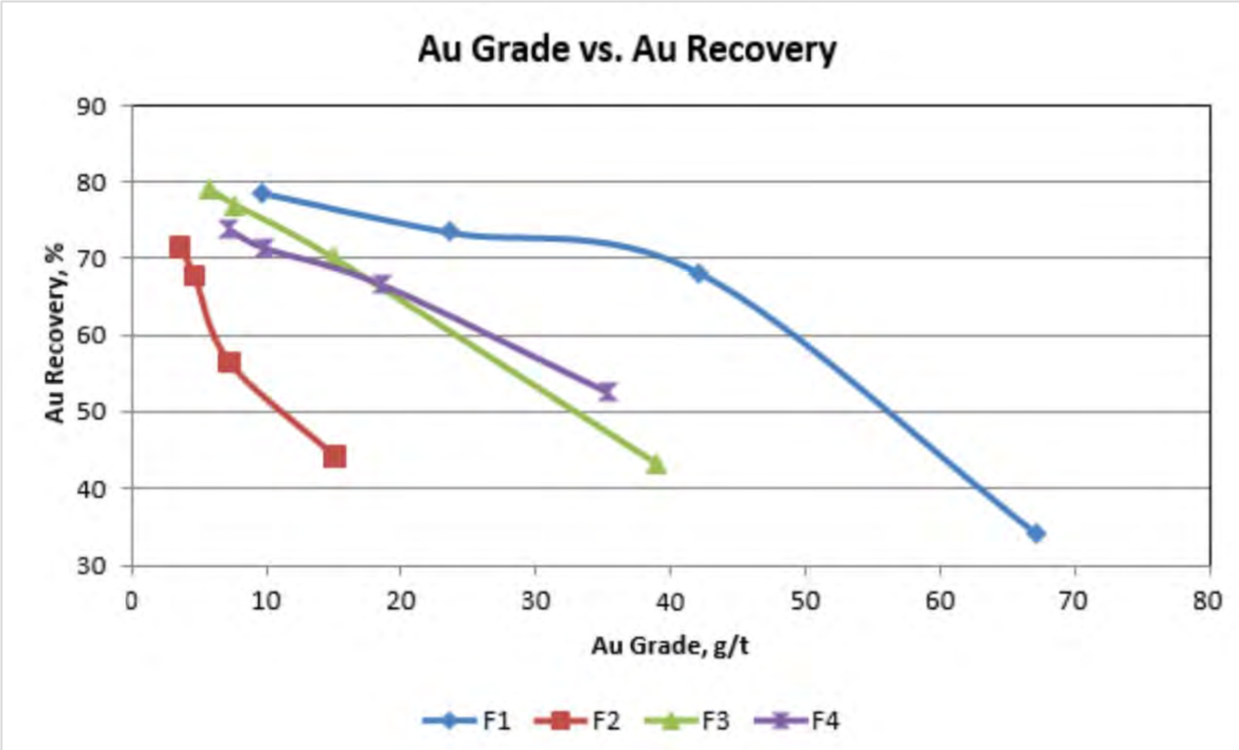
Both the coarse and fine saprolite samples were subjected to gravity concentration using a 3-inch Knelson® centrifugal concentrator followed by a Mozley table. The gravity test for the fine saprolite sample was performed on the sample without any prior processing. The coarse saprolite sample was ground to P_{80} 200 µm prior to the test. Au recoveries for the fine and coarse samples were approximately 50% and 27%, respectively. Intensive cyanide leaching of the gravity concentrates resulted in recoveries of 97% from both samples.

13.8.2 Flotation Testwork

Four flotation tests were performed on the fine saprolite sample to investigate different reagent schemes on the recovery of Au. Reagents used in the testwork included the following:

- Aerofloat 208;
- Aerofroth 5688;
- PAX;
- Max900; and
- MIBC.

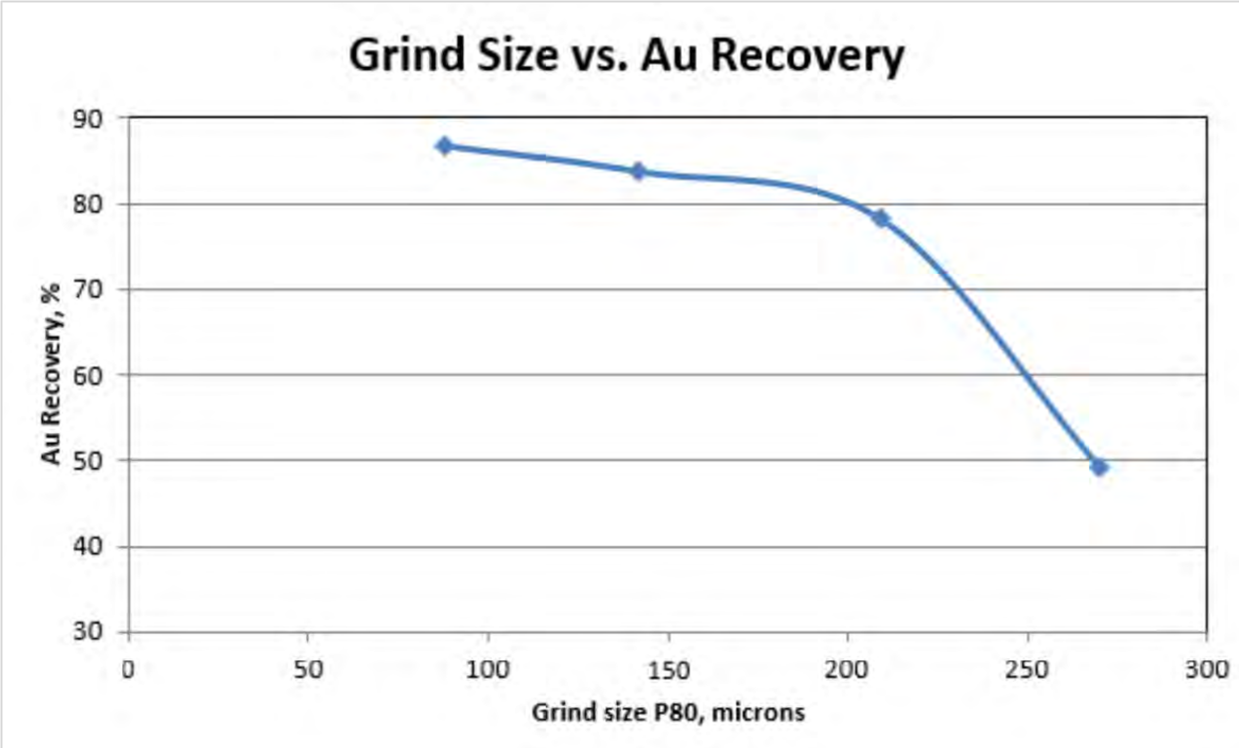
Recoveries for the fine saprolite sample were between 70% and 80% for all four tests. Grade vs. recovery curves for the four tests are shown in Figure 13-10.



Source: SGS, January 2013

Figure 13-10: Gold grade vs. gold recovery

Four additional tests were performed on the coarse saprolite sample to investigate the impact of grind size on the recovery of Au. The four different P_{80} grind sizes used were 270 µm, 209 µm, 142 µm, and 88 µm. Results showed that a decreasing grind size had a positive effect on the recovery of Au, with recoveries up to 86.7% for the finer grind size of 88 µm. The grind size vs. recovery curve for the tests is shown in Figure 13-11.



Source: SGS, 2013

Figure 13-11: Grind size vs. gold recovery

13.9 Cyanidation Testwork

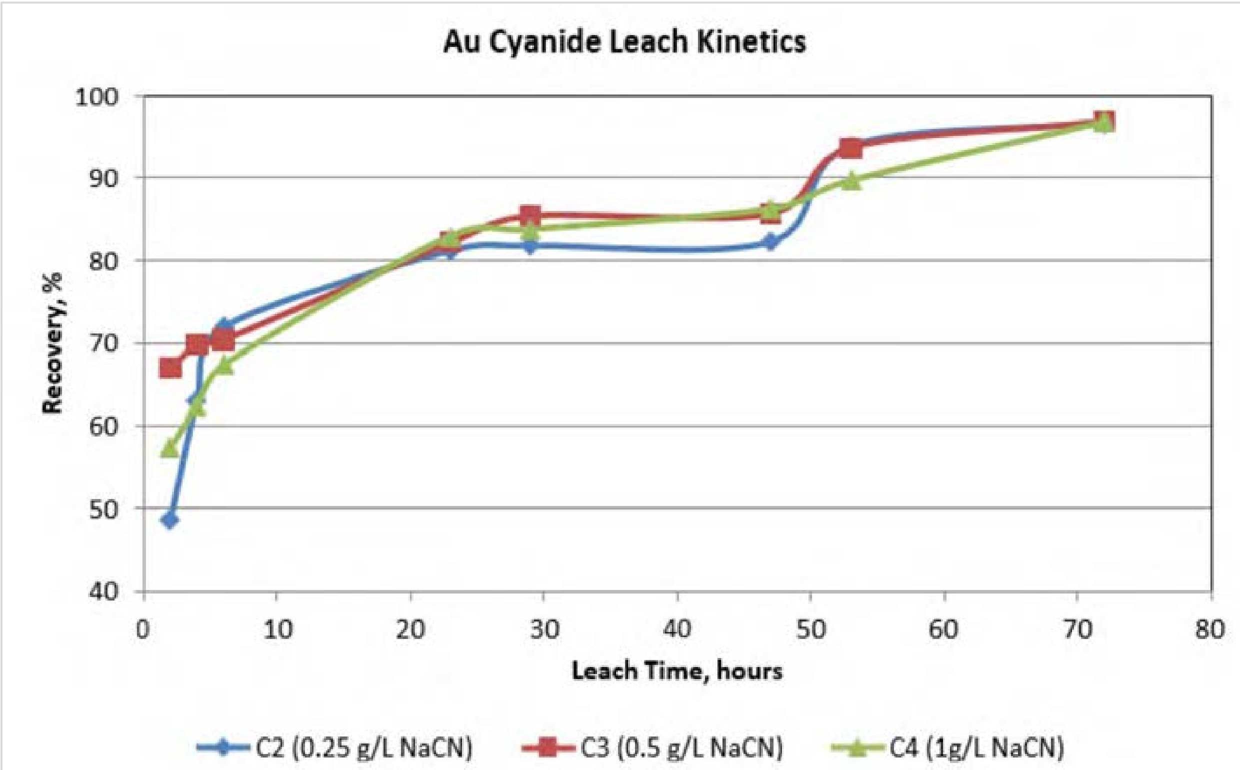
Cyanide leach and CIL tests were performed on the saprolite fines sample to investigate the effect of cyanide dosage in both leach and CIL circuits. Three separate NaCN dosages (0.25 g/l, 0.5 g/l, and 1.0 g/l) were tested on both the leach and CIL tests. Results from these tests showed that NaCN dosages have no effect on Au recovery, with all three dosages achieving the same recoveries. The leaching method did appear to affect recoveries, with the CIL tests achieving a slightly higher recovery of 98% versus the leach recovery of 96.8%. Tests results for the leach and CIL tests are shown in Table 13-50.

Table 13-50: Cyanide Leach at Different NaCN Dosages Test Summary

Test No	Sample ID	NaCN	Measured Head		Calculated Head		Leach Extraction		Residue		Consumption (kg/t)	
		g/L	Au (g/t)	Ag (g/t)	Au (g/t)	Ag (g/t)	Au(%)	Ag(%)	Au (g/t)	Ag (g/t)	NaCN	Ca $(OH)_2$
C2	Saprolite Fines	0.25	0.71	2.40	1.10	3.00	96.7	47.5	0.04	1.60	0.59	1.8
C3		0.50	0.71	2.40	0.90	3.60	96.8	44.2	0.03	2.00	1.15	1.4
C4		1.00	0.71	2.40	0.90	3.20	96.8	47.2	0.03	1.70	1.69	1.4
CIL1	Saprolite Fines	0.25	0.71	2.40	0.90	2.60	97.9	34.9	0.02	1.70	1.14	1.7
CIL2		0.50	0.71	2.40	1.00	1.50	98.1	54.6	0.02	0.70	1.77	1.4
CIL3		1.00	0.71	2.40	1.00	1.80	97.9	54.8	0.02	0.80	2.51	1.4

Source: Inspectorate, 2013

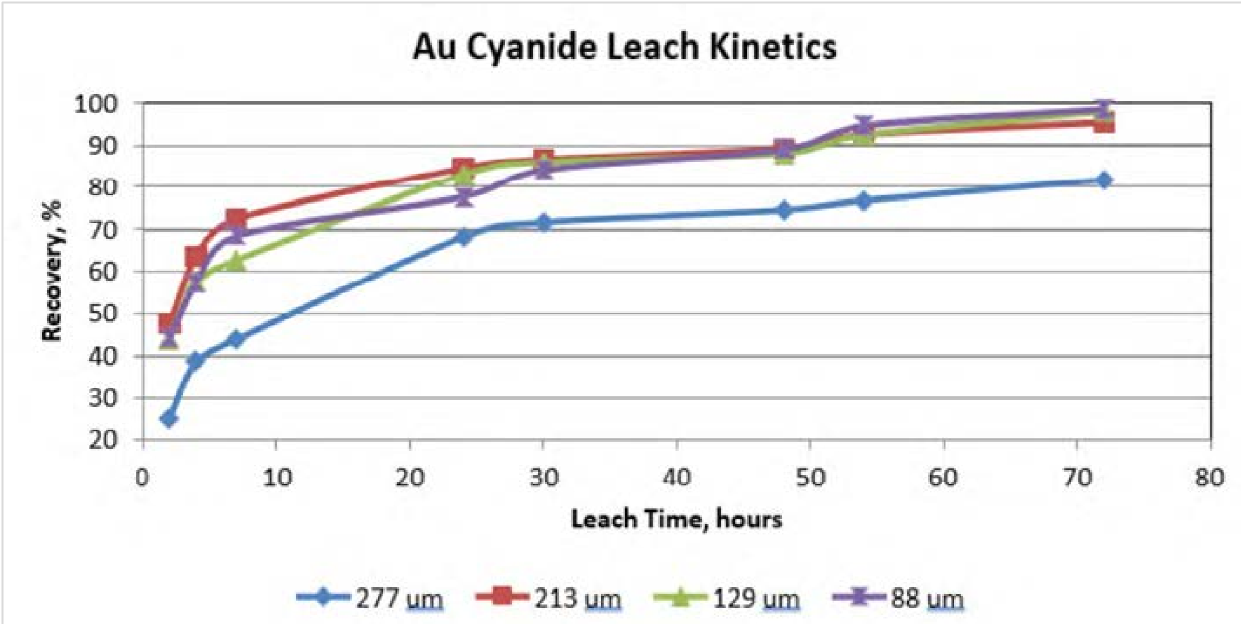
Intermediate samples were taken from the leach tests to examine the leaching kinetics of the sample. The tests showed that the leaching kinetics of the sample were slow, with the full 72 hours needed to approach maximum recoveries. Figure 13-12 shows the Au leaching kinetics curve for the sample.



Source: Tetra Tech, 2013

Figure 13-12: Au cyanide leach kinetics for saprolite fines

Cyanidation tests were also performed on the coarse saprolite sample to investigate the grind-recovery behaviour of the sample. Four different grind sizes (P_{80} 277 µm, 213 µm, 129 µm, and 88 µm) were tested. The tests show that the sample requires grinding to a P_{80} of at least 213 µm in order to facilitate acceptable recoveries. Grinding the sample from P_{80} 213 µm to P_{80} 129 µm showed slight improvement to the final recovery, indicating that the optimal grind size is between P_{80} 213 µm and P_{80} 129 µm. The grind vs. recovery curve for the samples is shown in Figure 13-13.



Source: Tetra Tech, 2013

Figure 13-13: Gold cyanide leach kinetics for coarse saprolite

One cyanide detoxification test was performed on the tailings from one of the CIL tests for the saprolite fines using the air/SO_2 process. Results from the test show that the air/SO_2 process can detoxify the tailings to achieve the required effluent standards. Results for this test are shown in Table 13-51.

Table 13-51: Cyanide Detoxification Test Results

Sample ID	Solution Analysis, mg/L								
	Tot (CN)	WAD (CN)	SO42-	Ag	As	Cu	Fe	Pb	Zn
Feed	83.3	80.59	52	0.02	<0.03	11.21	0.1	0.09	3.4
Detoxified sol	0.05	<0.05	3,069	<0.02	<0.03	0.08	<0.01	<0.05	<0.005

Source: Inspectorate, 2013

13.10 Tailings Settlement Tests 2012-2013

13.10.1 SGS Lakefield Test Program 2012/2013

Static settling tests were performed on the ACO-LCT Rougher Tailings from cycles A-C. These tests indicated the sample shows good flocculation and settling characteristics, the results of which are shown below in Table 13-52. These tests should be considered preliminary in nature as optimization of test conditions was not performed.

Table 13-52: ACO-LCT Rougher Tailings Cycles A-C Static Settling Test Results

Settling Test Number	1	2	3	4
Diluted Feed Solid Content, % wt.	5	10	15	20
CIBA Magnafloc 333, g/t	12	12	12	12
U/F Solids Density, % wt	61	71	71	70
Thickener Underflow Unit Area m^2/tpd	*	0.058	0.06	0.065
Thickener Hydraulic Unit Area m^2/tpd	969	0.012	0.012	0.015
Initial Settling Rate, m^3/m^2/day	969	714	438	241

Source: SGS, 2013. * Due to extremely high settling rates the compression point could not be measured with sufficient accuracy in order to allow for the calculation of thickener specific unit areas.

13.10.2 Inspectorate Test Program 2012 – 2013

Scoping level tests were performed on detoxified saprolite tailings to identify potential candidates for flocculant and to examine the settling characteristics. These tests indicated that the detoxified saprolite tailings have a poor flocculation response. Additionally, a two-litre test performed over the course of six days indicated that an underflow density of 52.6% could be achieved with a clear overflow solution without flocculant as part of a tailings pond study.

13.11 Metallurgical Test Work Program 2014

13.11.1 General

The initial 2014 metallurgical test work program was conducted at FLSmidth Dawson Metallurgical Laboratory. This included the comminution, gravity and sedimentation test work, and some initial scouting test work for flotation and leaching.

The detailed flotation and leaching testwork was transferred to ALS Metallurgical Laboratories (Kamloops) due to resource constraints at FLSmidth Dawson.

13.11.2 FLS Gravity Test Work

A single bulk composite of both ACO and LCO material was prepared from existing available samples and was subjected to the standard E-GRG (Enhanced Gravity Recoverable Au) test which requires stage grinding and gravity recovery of the material.

The targeted grind for the Project has been determined elsewhere and has been set at 150 µm. This grind target was used to correct the liberated gravity recoverable Au from the results of the E-GRG tests. In this way, it was determined that the corrected gravity recoverable Au value for the sample at a grind of 150 µm, was 38.5%.

Circuit modelling conducted by FLSmidth Knelson indicated that the optimum circuit would consist of two Knelson concentrators per milled stream treating approximately 30% of the cyclone underflow. Under these conditions, it would be expected that approximately 14.5% of the mill feed Au would be recovered to gravity concentrate.

13.11.3 FLS Flotation Test Work

A preliminary test program was conducted during February and March of 2014 at FLSmidth Dawson Metallurgical Laboratory to identify initial flotation test parameters.

Six kinetic flotation tests at a target grind P_{80} of 125 µm were conducted to evaluate the use of Cu selective collectors (Cytec 7017, Flomin 4132 and Cytec A-208) to produce a high Cu grade rougher concentrate, followed by a bulk rougher scavenger float (using PAX) to maximize the Au, Ag and Cu recovery. Test material provided by the Company represented the two predominant lithological variations in the deposit that contained higher Cu grades, known as ACO. Each lithology sample was crushed, ground, was subjected to gravity separation to remove gravity Au, and was then split into three identical samples for flotation testing.

The three Cu selective reagents dosed at 10 g/t produced comparable results after three minutes of rougher flotation, with Cu grades between 15% and 16% and an average recovery of 86%, Au grades between 43 g/t and 48 g/t and an average recovery of 75% and Ag grades between 132 g/t and 143 g/t and an average recovery of 63%.

A further 18 minutes of rougher scavenger flotation using PAX as a collector increased overall flotation recovery to approximately 97% Cu, 86% Au, 78% Ag and 98% total sulphur.

Further test work was carried out using varying doses of Cytec 7017 to determine if a high-grade final Cu concentrate could be obtained in the rougher circuit. The dose was varied from 2.5 g/t to 10 g/t. No improvement in Cu grade was achieved, and Cu recoveries dropped off at the lower collector doses.

13.11.4 ALS Flotation (and Flotation Product Cyanide Leaching)

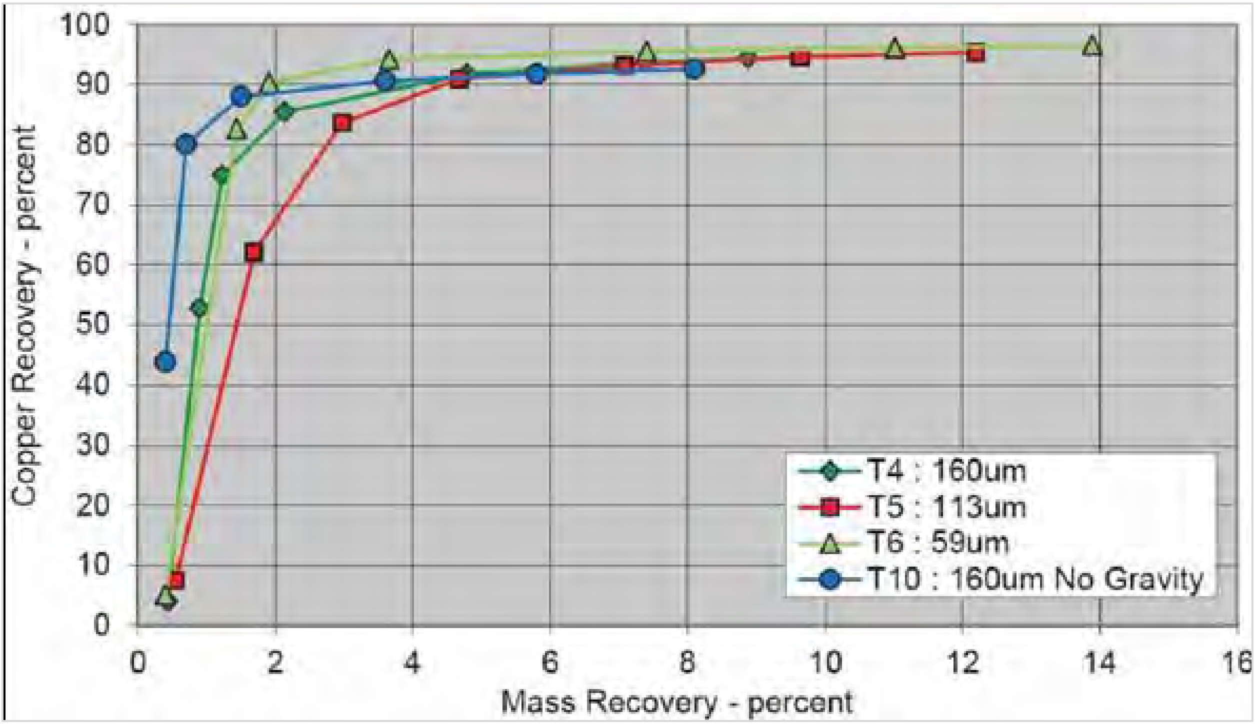
The majority of the 2014 flotation test work program was conducted by ALS Kamloops.

A series of batch rougher and cleaner tests were conducted on the Bulk Copper Composite to develop a flowsheet and reagent scheme for the Cu phase of the Project. Testing culminated in LCTs on the two Locked Cycle composites, simulating the closed circuit performance of the flowsheet. The developed flowsheet was then applied to the Variability and SC.

Roughing tests were conducted on the Bulk Copper Composite at P_{80} grinds of 59, 113 and 160 µm utilizing Cytec 7017 collector plus PAX in the latter (scavenging) stages of the tests. The mineralized materials natural pH of 8.8 8.9 was used along with MIBC as a frother.

Grinds finer than P_{80} of 160 µm did not result in a significant improvement in Cu Au or Ag recovery. Finer grinds were considered sub economic but there is a case for coarser grinds to be applied which was not explored as part of the program.

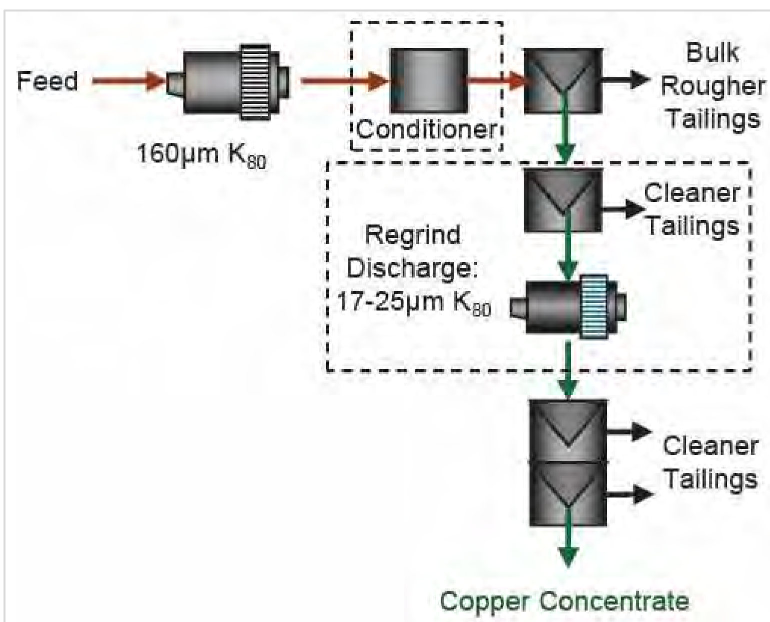
A gravity stage on the flotation flowsheet was not found to increase the overall Au or Ag recovery. However, the return on Au (revenue) from a gravity concentrate would probably be greater than the losses attributed to the Au sales in a mixed concentrate. Results are summarized by Figure 13-14.



Source: ALS, January 2015

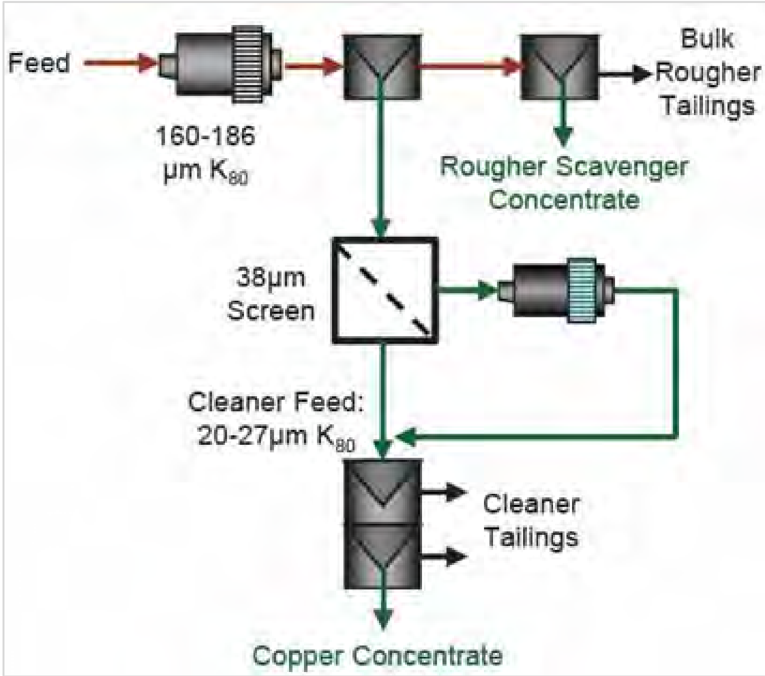
Figure 13-14: Rougher flotation performance bulk copper composite

Two different cleaning flowsheets were used to assess cleaning performance on the Bulk Copper Composite rougher concentrates. Refer to Figure 13-15 and Figure 13-16 which present the options graphically.



Source: ALS, January 2015

Figure 13-15: Cleaning flowsheet 1



Source: ALS, January 2015

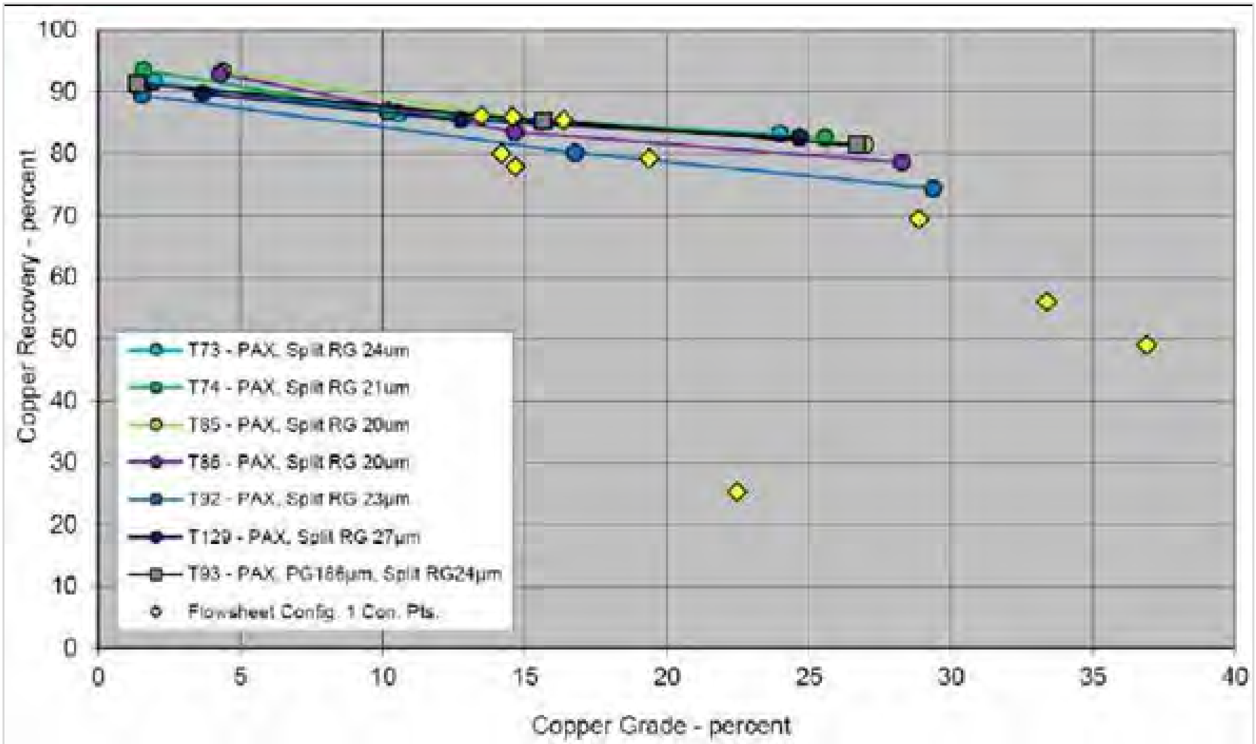
Figure 13-16: Cleaning flowsheet 2

Various grinds and reagent regimes were trialled. Cleaner tests using the first configuration were conducted at a primary grind sizing of 160 µm P_{80} based on rougher test results. The flowsheet utilized Cytec 7017 as the primary collector with additional PAX or 3418A as supplementary collectors; two tests were conducted with PAX as the sole primary collector. A conditioning stage was required for the use of Cytec 7017 but was not required for PAX.

The effects of regrinding were investigated. When regrinding was employed, a stage of dilution cleaning was added prior to the regrind to reduce the mass requiring regrinding. Regrind discharge sizings ranged from 17 to 25 µm P_{80}.

The second cleaning configuration also employed a primary grind sizing of 160 µm P_{80} with one test being conducted at about 186 µm P_{80}. The flowsheet utilized PAX as the sole collector. Prior to a regrinding stage, the rougher concentrate was screened using a 38 µm screen and the fraction coarser than 38 µm being reground. The fines and the reground material were recombined to form the cleaner circuit feed. The intent was to create a size distribution that was more representative of a regrind operation in closed circuit with a cyclone. Cleaner feed sizings during testing of this flowsheet configuration ranged from 20 to 27 µm P_{80}. An addition of a rougher scavenger stage was tested for cyanidation leach testing on the rougher scavenger concentrate.

The optimal test results, in terms of Cu performance, utilized PAX and the split regrind configuration. Refer to Figure 13-17 for a summary of performance.



Source: ALS, January 2015

Figure 13-17: Split regrind cleaning performance on bulk copper composite split regrind cleaning flowsheet performance on bulk copper composite concentrate.

Copper concentrates using this configuration averaged 27% Cu at a Cu recovery of around 80%. Au recovery to these concentrates averaged around 67% and graded, on average, around 161 g/t Au. Ag recovery averaged about 60%, and graded, on average, 314 g/t Ag. This was achieved with regrind discharge sizings of between 20 and 27 µm P_{80}. One test (T56) with PAX employed regrinding but not the split regrind configuration, and poorer Cu performance was measured. When the primary grind sizing was coarsened to 186 µm P_{80} the difference in performance was negligible. This suggests it may be beneficial to conduct further testing at a coarser grind to determine if primary grinding requirements could be lessened without additional Cu and Au losses.

All cleaner testing conducted without regrind showed poor upgrading potential. Copper grades above about 19% were not achieved.

Cleaner testing using Cytec 7017 yielded poorer Cu performance, as a significant portion of the Cu was lost following regrinding. Ag and Au were not lost following the regrind and achieved comparable, if not better performance compared to using PAX with the split regrind

13.11.5 Variability

The flowsheet and conditions developed through testing on the Copper Bulk Composite were applied to the two Locked Cycle composites, two Variability Composites, the Float Variability Composite and the four SC. Table 13-53 presents performance results from this work.

Table 13-53: Variability Composite Batch Results

Composite	Test	Assay % or g/t			Recovery %			Feed Grade % or g/t			Additional Test Variable
		Cu	Ag	Au	Cu	Ag	Au	Cu	Ag	Au	
Cu Bulk	T129	24.7	288	144	82.6	64.3	64.3	0.15	2	1.13	Optimized Baseline Test
Cu GradeVar 1	T130 T137	20.1 25.4	216 304	138 200	80.8 69.0	56.2 50.6	68.2 62.0	0.09 0.09	1 1	0.74 0.76	None None
Cu GradeVar 3	T131 T138 T145	29.4 30.4 26.9	332 362 320	89.0 140 104	85.6 88.2 85.5	72.5 79.0 72.8	57.6 74.7 68.2	0.34 0.34 0.37	5 5 5	1.53 1.88 1.78	None Increased Cleaner Collector Repeat Test 138
Float Variability	T146	27.1	262	154	81.5	68.0	70.0	0.15	2	1.02	None
Cu Spatial 1	T151	23.8	176	76.1	86.3	62.0	72.6	0.20	2	0.74	None
Cu Spatial 2	T152	24.4	264	110	89.4	69.3	68.8	0.20	~3	1.17	None
Cu Spatial 3	T153	27.5	236	124	82.2	49.0	67.7	0.22	3	1.22	None
Cu Spatial 4	T132	22.4	198	129	85.3	56.2	56.1	0.19	3	1.65	None
Cu LockedCycle 1	T141	25.2	278	129	78.2	60.1	59.4	0.12	2	0.84	None
Cu LockedCycle 2	T133 T139	31.8 26.7	414 354	163 173	73.5 83.8	57.0 67.4	51.3 75.9	0.15 0.15	3 2	1.10 1.07	None Increased Cleaner Collector

Source: ALS, 2015. Copper assays are as percent. Au and Ag assays are in g/t.

Performance of the various composites was similar to that of the Copper Bulk Composite in batch cleaner testing. Between 69% and 89% of the tested Cu was recovered to the Cu concentrate, which graded between 20% and 32% Cu. Ag recovery to the concentrate ranged from about 49% to 79% and Au recoveries ranged from about 51% to 76%.

The Variability Composites showed the greatest variation in performance from the other composites. The Copper Grade Variability 1 Composite graded the lowest in feed Cu content at about 0.09% and performance was the poorest, while the Copper Grade Variability 3 Composite measured the highest feed Cu content and showed the best performance. The results showing what may be an anticipated grade response to such testing.

13.11.6 Locked Cycle Testing

Locked cycle testing was conducted on several of the composites for assessment of closed circuit performance as well as for production of stream products that could be used in subsequent testing. The locked cycle testing involved the recirculation of the second cleaner tailing stream into the cleaner feed of the subsequent cycle to determine the effects of recirculation on performance. Table 13-54 summarizes the results.

Table 13-54: Results of LCTs

Product	Weight %	Assay % or g/t				Distribution %			
		Cu	Ag	Au	S(t)	Cu	Ag	Au	S(t)
Cu Locked Cycle 1 Composite Test 143: Cycle IV + V									
Flotation Feed	100.0	0.13	2	1.03	0.13	100.0	100.0	100.0	100.0
Cu Con	0.4	27.6	306	195	25.9	78.6	72.2	68.9	69.8
Cu 1st Clnr Tail	4.5	0.36	4	3.05	0.40	12.7	11.7	13.4	13.4
Cu Ro ScavCon	4.4	0.06	<1	0.64	0.05	2.0	1.4	2.7	1.7
Cu Ro ScavTail	90.7	0.01	<1	0.17	0.02	6.7	14.7	15.0	15.1
Cu Locked Cycle 2 Composite Test 144: Cycle IV + V									
Flotation Feed	100.0	0.15	3	1.15	0.20	100.0	100.0	100.0	100.0
Cu Con	0.4	26.5	359	162	27.7	78.6	61.7	62.4	62.8
Cu 1st Clnr Tail	2.5	0.59	14	6.58	0.90	9.9	13.6	14.4	11.6
Cu Ro ScavCon	3.8	0.21	5	2.82	0.26	5.3	6.6	9.3	5.1
Cu Ro ScavTail	93.3	0.01	<1	0.17	0.04	6.2	18.0	13.9	20.5
Float Variability Composite Test 147: Cycle IV + V									
Flotation Feed	100.0	0.17	2	1.32	0.23	100.0	100.0	100.0	100.0
Cu Con	0.6	24.0	252	169	29.6	88.6	68.7	79.8	79.8
Cu 1st Clnr Tail	3.2	0.20	3	3.12	0.49	3.8	4.9	7.6	6.8
Cu Ro ScavCon	3.0	0.16	3	0.86	0.19	2.9	4.0	2.0	2.5
Cu Ro ScavTail	93.1	0.01	<1	0.15	0.03	4.7	22.4	10.6	10.9
Float Variability Composite Test 148: Cycle IV + V									
Flotation Feed	100.0	0.17	2	1.27	0.22	100.0	100.0	100.0	100.0
Cu Con	0.5	27.3	273	177	31.2	87.2	65.4	74.2	75.9
Cu 1st Clnr Tail	3.5	0.29	6	2.72	0.71	6.0	10.2	7.5	11.3
Cu Ro ScavCon	3.3	0.14	2	0.72	0.15	2.7	3.5	1.9	2.2
Cu Ro ScavTail	92.7	0.01	<1	0.23	0.02	4.2	20.8	16.4	10.6
Spatial Composite Blend Test 154: Cycle IV + V									
Flotation Feed	100.0	0.21	3	1.30	0.31	100.0	100.0	100.0	100.0
Cu Con	0.8	23.6	232	125	31.4	89.8	63.0	75.4	80.0
Cu 1st Clnr Tail	4.2	0.22	7	2.84	0.41	4.6	10.2	9.3	5.7
Cu Ro ScavCon	3.4	0.12	3	1.20	0.17	2.0	3.6	3.2	1.9
Cu Ro ScavTail	91.6	0.01	<1	0.17	0.04	3.6	23.2	12.2	12.4

Source: ALS, 2015. Copper assays are as percent. Au and Ag assays are in g/t.

Overall, the locked cycle testing showed results which are typical of closed circuit performance in that recovery of Cu was higher than batch test performance with slightly lower concentrate grades compared with the batch test maxima. Copper recoveries to the concentrate ranged from 79% to 90%, while the Cu concentrate grades ranged from around 24% to 28%. Ag and Au upgraded quite well to the Cu concentrate grading between 230 g/t and 360 g/t for Ag and between about 125 g/t and 195 g/t for Au. Ag recoveries ranged from 62% to 72% and Au recoveries ranged from 62% to 80%.

Lower Cu recoveries were observed from the two Locked Cycle composites. The majority of these higher losses were seen through the first cleaner tailing stream. There may be a potential for the incorporation of a cleaner scavenger stream into the circuit to improve cleaner circuit recovery.

The rougher scavenger concentrate was included for subsequent cyanidation leach testing.

13.11.7 Copper Concentrate Quality

Several Cu concentrates were analyzed through multi-element ICP analyses to determine the levels of possible deleterious and trace level elements. In addition, two concentrates from locked cycle testing were submitted for QEMSCAN Bulk Mineral Analyses (BMA) to determine the mineral content of the samples.

Several deleterious elements were present in the concentrates at concentrations that may invoke penalty upon sale including bismuth, selenium, and tellurium. Bismuth ranged from nominally 380 g/t to 631 g/t in the concentrates. Selenium ranged from around 500 g/t to 770 g/t in the concentrates, and tellurium ranged from 292 g/t to 396 g/t in the concentrates. These minerals upgraded very similarly to Ag suggesting a potential association.

Two concentrates underwent mineralogical assessments for mineral content. Approximately 25% of the concentrate mass for each concentrate was comprised of non sulphide gangue minerals, mainly silicates. This suggests finer regrinding or additional dilution cleaning may aid in improving concentrate grade. This would also likely upgrade deleterious elements. The results are presented in Table 13-55.

Arsenic content measured close to typical penalty levels and could increase above these levels should more of the non sulphide gangue be rejected. A full QEMSCAN Particle Mineral Analysis (PMA) determining mineral interlocking would be required to confirm the potential behaviour and performance opportunities.

Table 13-55: Concentrate Quality Mineral Deportment

Minerals	Test 143 Copper Concentrate I-V	Test 144 Copper Concentrate I-V
Chalcopyrite	56.3%	50.6%
Bornite	15.1%	14.1%
Chalcocite	0.2%	0.1%
Covellite	<0.1%	<0.1%
Enargite/Tennantite	0.2%	<0.1%
Molybdenite	0.3%	0.2%
Pyrite	3.1%	9.9%
Non sulphide Gangue	24.7%	24.9%

Source: ALS, 2015. Non sulphide gangue includes quartz, feldspars, epidote, chlorite, micas, and various other minerals.

Each composite was low temperature dried, bigger lumps were broken by rolling without crushing and then thoroughly blended by riffling three times before splitting into test charges and head assay aliquots. Triplicate splits from each sample were pulverized and assayed for Au, Ag, S speciation, C speciation, Hg and ICP. Another triplicate split of ~500 g from each sample was subjected to metallic screen. A total of 191 kg of saprolite composite was prepared, and a total of 68 kg of Transition composite was prepared.

Several cyanide bottle roll tests were conducted over 48 hours on two of the exit stream products from batch cleaner testing. Leached were the rougher scavenger concentrate and the first cleaner tailing. Table 13-56 and Table 13-57 provide a summary of the conditions and results from the cyanide leach extraction tests on these streams.

Table 13-56: Rougher Scavenger Concentrate Cyanide Leaching

Source Composite	Cyanide LeachTest	FloatTest	Recovery to Ro. Scav Con %		Au Recovery		Au Grade	g/t	Reagent Consumption			
									kg/t Ro. Scav. Con		kg/t Flot Feed	
			Mass	Au	% of Leach Feed	% of Float Feed	Leach Feed	Residue	NaCN	Lime	NaCN	Lime
Cu Bulk Composite	82	73	1.5	2.4	78.0	1.9	1.23	0.27	1.8	2.9	0.03	0.04
	84	74	2.8	1.5	99.2	1.5	1.22	0.01	1.0	1.8	0.03	0.05
	90	85	2.2	1.2	98.2	1.2	0.57	0.01	0.6	1.3	0.01	0.03
	91	86	5.9	2.2	80.3	1.8	0.51	0.10	0.5	0.8	0.03	0.05
Float Var	149	147/148	3.2	1.9	62.5	1.2	0.85	0.32	1.7	1.1	0.05	0.03
Cu Locked Cycle 1	155	143	4.4	2.7	73.7	2.0	0.59	0.16	0.7	0.8	0.03	0.03
Cu Locked Cycle 2	156	144	3.8	9.3	30.5	2.8	1.91	1.33	1.2	1.2	0.05	0.05
Spatial Blend	157	154	3.4	3.2	58.4	1.8	0.94	0.39	0.7	1.2	0.03	0.04

Source: ALS, 2015

Table 13-57: First Cleaner Tail Cyanide Leaching

Source Composite	Cyanide LeachTest	FloatTest	Recovery to First Cleaner Tail %		Au Recovery		Au Grade	g/t	Reagent Consumption			
									kg/t 1st Clnr Tail		kg/t Flot Feed	
			Mass	Au	% of Leach Feed	% of Float Feed	Leach Feed	Residue	NaCN	Lime	NaCN	Lime
Cu Bulk Composite	81	73	4.3	7.0	68.7	4.8	1.84	0.58	3.4	3.8	0.15	0.16
	83	74	5.0	7.5	76.8	5.8	1.86	0.43	3.7	1.3	0.18	0.07
	101	*	5.0	3.8	76.8	2.9	0.70	0.23	1.6	0.7	0.08	0.03
Float Var	150	147/148	4.7	12.1	49.5	6.0	2.61	1.32	1.9	1.2	0.09	0.06

Source: ALS, 2015

Cyanide leaching of the rougher scavenger concentrates recovered between 31% and 80% of the remaining Au in tests without regrinding and between 98% and 99% of the remaining Au in tests with regrind. This accounted for about 1% to 3% of the Au in the flotation feed. The difference in recovery between tests with and without regrind may be due to differences in the nature of the Au in the leach feed versus regrinding itself. Further repeat testing would be required to confirm the improvement from regrinding.

The 1st cleaner tail stream from Tests 73 and 74 contained between 7% and 8% of the Au from the Copper Bulk Composite in around 4% to 5% of the mass. Between 69% and 77% of this Au was extracted during the tests, representing between 5% and 6% of the feed Au.

A composite of first cleaner tailings was formed from various batch cleaner tests for additional cyanide leach extraction testing. This composite represented about 5% of the mass and 4% of the Au from the Copper Bulk Composite. Cyanidation leaching resulted in extraction of about 77% of this Au, representing 3% of the feed Au.

A cleaner tailing sample from the cycle testing on the Float Variability Composite was also cyanide leached. This sample contained some 12% of the feed Au, of which approximately 50% was extracted over the 48-hour test.

13.11.8 FLS Cyanidation Test Work

A preliminary test program was conducted February and March of 2014 at FLSmidth Dawson Metallurgical Laboratory to identify initial leach test parameters.

Six, 48-hour kinetic leach tests were conducted to evaluate the impact that grind would have on overall Au recovery. Test material provided by the Company represented the two predominant lithological variations in the mineralized material body that contained low Cu grades, known as LCO. Each lithology sample was crushed and split into three sub samples. These in turn were ground to a P_{80} of 75 µm, 125 µm and 150 µm respectively. Each ground sample was subject to gravity separation and then leached for 48 hours using 1 g/l NaCN and maintaining a slurry pH of 10.8 to 11.0.

LCO-V Gravity Au recovery appears to peak at around 120 µm, while both leach and overall recovery steadily improve with finer grind. LCO-I Au recovery for gravity, leach, and overall peaks at 120 µm. The two lithologies respond very differently to both gravity and leach processes, with the volcanic lithology showing significantly higher recoveries over the intrusive lithology. The Au recovery is presented in Table 13-58.

Table 13-58: Summary of Mineralogy Results

Sample Source	Particle Size	Au Recovery (%)		
	P_{80}	Gravity	Leach	Overall
LCO-V	148	52.1	89.5	95.0
LCO-V	122	63.5	92.6	97.3
LCO-V	70	49.3	94.5	97.2
LCO-I	160	18.2	89.0	91.0
LCO-I	123	25.3	93.1	94.9
LCO-I	78	22.3	91.4	93.3

Source: FLS, 2014

13.11.9 ALS Cyanide Test Work (Fresh Mineralized Materials)

13.11.9.1 Gold Composite Testing

A Gold Bulk Composite, a Gold Variability Composite and four Gold SC were prepared for the purpose of developing an Au extraction flowsheet involving gravity concentration and cyanide leach extraction. A series of chemical and mineralogical analyses were conducted on these composites followed by metallurgical testing to develop and optimize a flowsheet. The following subsections summarize the results from the analysis and testing.

13.11.9.2 Chemical and Mineral Content

A suite of chemical assays was completed on the Gold composites, along with BMA to determine the mineral content of the samples. Chemical content is summarized in Table 13-59, while mineral content results are summarized in Table 13-60.

Table 13-59: Chemical Content Summary

Composite	Assay % or g/t							
	Cu	CuCN	S(s)	Ag(t)	Au	AuSM	C	TOC
Au Bulk	0.05	0.002	0.10	<1	1.19	1.08	0.77	0.02
Au Spatial 1	0.06	0.003	0.06	<1	0.75	0.99	0.81	0.02
Au Spatial 2	0.04	0.002	0.10	<1	1.01	1.98	0.77	0.02
Au Spatial 3	0.05	0.002	0.11	<1	0.67	112	0.78	0.02
Au Spatial 4	0.06	0.004	0.21	<1	0.87	0.92	0.86	0.02
Au Variability	0.10	0.029	-	-	1.01	-	-	-

Source: ALS, 2015

- Ag and Au assays are displayed in g/t; other assays are displayed in percent.
- CuCN Cu soluble in a weak NaCN solution: S(s) sulphur contained in sulphide minerals: Ag(t) total Ag content, by multi-acid digestion: AuSM Au content determined through the screened metallic assay method: TOC Total organic carbon.

Table 13-60: Mineral Content Summary

Mineral	Content %				
	Au Bulk Composite	AuSpatial 1	AuSpatial 2	AuSpatial 3	AuSpatial 4
Copper Sulphides	0.1	0.1	0.1	0.2	0.1
Sphalerite	-	<0.1	-	-	<0.1
Pyrite	0.1	0.1	0.3	0.2	0.3
Quartz	33.1	30.6	34.6	28.0	32.5
Feldspars	31.9	33.0	30.8	31.4	31.6
Micas	11.4	7.4	9.2	9.5	11.8
Chlorite	12.5	14.4	12.3	13.6	11.2
Calcite	2.4	3.3	2.8	3.2	3.2
Epidote	1.9	2.1	2.4	2.6	2.1
Ti-Minerals	1.6	2.0	1.8	2.3	1.7
Amphibole	2.2	3.9	2.8	5.0	2.7
Biotite Phlogopite	1.2	1.5	1.4	1.9	1.4
Others	15	1.7	1.6	2 1	1.3

Source: ALS, 2015

The Gold Bulk Composite assayed about 1.1 g/t Au using the screened metallic method. Across the SC, this assay varied from 0.9 g/t to about 2.0 g/t; the screened metallic Au assay was typically higher than the fire assay method.

Sulphur measured in sulphide form averaged about 0.1% and ranged from about 0.1% to 0.2% across the SC. This was contained in pyrite and the Cu sulphide chalcopyrite. Only trace levels of Cu assayed as cyanide soluble; higher content of cyanide soluble Cu would increase the cyanide consumption during a cyanide leach.

While carbon assayed between 0.8% and 0.9% in the composites, only trace amounts assayed as organic carbon. Organic carbon can remove solubilized Au from solution during a cyanidation leach, reducing Au recovery.

13.11.9.3 Metallurgical Testing

A series of Knelson Gravity concentration tests were carried out on the composites at varying grind sizing with cyanidation bottle roll leaches of the gravity tailings at various leach conditions to assess the effect upon Au and Ag extraction. The following sections discuss the results of this testing.

13.11.9.4 Gold Bulk Composite Testing

The optimal conditions determined through testing on the Gold Bulk Composite were at a primary grind of 162 µm P_{80}, with a pre-aeration stage, using a cyanide concentration of 500 ppm sodium cyanide (NaCN), a leach pH of 11 and with oxygen sparging throughout. At these conditions, about 56% of the Au was recovered to the gravity concentrate and a further 36% to 37% of the Au was extracted to the leach liquor from the gravity tailings; this resulted in a total Au extraction of between 92% and 93%. Reagent consumptions were calculated at about 0.2 kg of NaCN and about 0.6 to 0.7 kg of lime per tonne of feed.

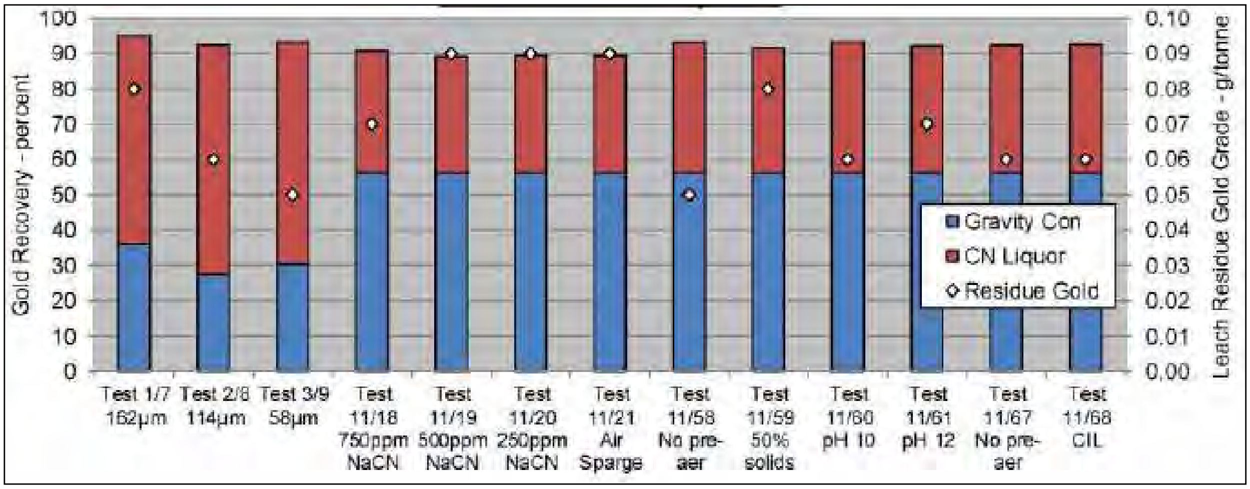
Similar extractions were seen from the CIL test as from the standard bottle roll tests without carbon. Higher cyanide consumptions were recorded at about 0.4 kg NaCN per tonne feed and lower lime consumptions at about 0.3 kg lime per tonne feed. Test conditions are summarized by Table 13-61 and Figure 13-18.

Table 13-61: Test Conditions and Result Summary Gold Bulk Composite

Test# Grav/CN	Grind Size P_{80}	NaCN Conc, ppm	pH	Pulp Density %	Extraction %			Residue Au g/t	Reagent Cons, kg/t Feed	
					Gravity	Leach	Overall		NaCN	Lime
1/7	162	500	11	33	36.1	58.9	95.0	0.08	0.2	0.4
2/8	114	500	11	33	27.4	65.1	92.5	0.06	0.2	0.3
3/9	58	500	11	33	30.3	63.0	93.2	0.05	0.3	0.4
11/18	162	750	11	33	56.3	34.5	90.8	0.07	0.4	0.5
11/19	162	500	11	33	56.3	32.8	89.0	0.09	0.2	0.6
11/20	162	250	11	33	56.3	33.2	89.5	0.09	0.2	0.6
11/21	162	500	11	33	56.3	33.1	89.4	0.09	0.2	0.6
11/58	162	500	11	33	56.3	36.9	93.1	0.05	0.2	0.6
11/59	162	500	11	50	56.3	35.3	91.6	0.08	0.2	0.6
11/60	162	500	10	33	56.3	37.0	932	0.06	0.2	0.1
11/61	162	500	12	33	56.3	35.9	922	0.07	0.1	2.4
11/67	162	500	11	33	56.3	36.1	92.3	0.06	0.2	0.7
11/68	162	500	11	33	56.3	36.2	92.5	0.06	0.4	0.3

Source: ALS, 2015

- Ag assays of products were near or below detection limits, limiting accuracy of results for Ag performance.
- Test 11 gravity extraction refers to the Au recovery through cyanide leaching of the gravity concentrate in Tests 12,13 and 14.
- Test 68 was a CIL bottle roll test.
- Test 21 was sparged with air as opposed to oxygen as in other tests.



Source: ALS, 2015

Figure 13-18: Leach result summary gold bulk composite

13.11.9.5 Gold Spatial Composite Testing

The cyanidation leach conditions developed in testing on the Gold Bulk Composite were also applied to the four Gold SC. The conditions were applied at four primary grind sizings. Table 13-62 displays the summary of results at the optimal grind sizing established through testing on the Au Bulk Composite. Table 13-63 displays a summary of the test results under varying conditions.

Table 13-62: Optimal Condition Result Summary Gold SC

Test	Sizing *** µm P_{80}	Recovery %			Reagent Cons kg/t **	
		Gravity	Leach	Total	NaCN	Lime
69 *	189/159	37.5	59.7	97.1	0.2	0.5
70	171	19.4	71.1	90.5	0.1	0.4
71 *	212/151	21.2	68.9	90.1	0.1	0.4
72 *	195/164	40.3	53.0	93.3	0.1	0.4

Source: ALS, 2015

* Gravity and leach extractions were conducted at different grind sizes.
**Consumptions are in relation to leach feed mass.
*** Second number refers to the leach feed sizing if different than the gravity.

Table 13-63: Test Condition Result Summary Gold SC

Composite	Test# Grav/CN	Leach Feed Size µm P_{80}	Extraction Percent			Residue Au	Reagent Cons, kg/t *	
			Gravity	Leach	Overall	g/t	NaCN	Lime
Au Spatial 1	63/69	159	37.5	59.7	97.1	0.04	0.2	0.5
	31/43	135	34.6	61.4	95.9	0.03	0.2	0.6
	32/44	129	37.8	57.6	95.5	0.05	0.3	0.6
	33/45	50	49.4	48.1	97.5	0.02	0.3	0.7
Au Spatial 2	34/46	176	38.6	53.0	91.5	0.12	0.2	0.6
	64/70	171	19.4	71.1	90.5	0.10	0.1	0.4
	35/47	125	322	58.7	90.9	0.10	0.2	0.7
	36/48	64	352	58.6	93.8	0.05	0.3	0.7
Au Spatial 3	37/49	169	22.9	68.3	91.3	0.05	0.3	0.6
	65/71	151	212	68.9	90.1	0.07	0.1	0.4
	38/50	131	35.0	59.4	94.4	0.05	0.2	0.6
	39/51	62	41.5	55.5	97.0	0.02	0.3	0.7
Au Spatial 4	66/72	164	40.3	53.0	93.3	0.08	0.1	0.4
	40/52	139	46.5	47.8	94.2	0.04	0.2	0.5
	41/53	119	49.8	46.5	96.2	0.04	0.2	0.6
	42/54	73	58.7	38.1	96.7	0.03	0.3	0.6

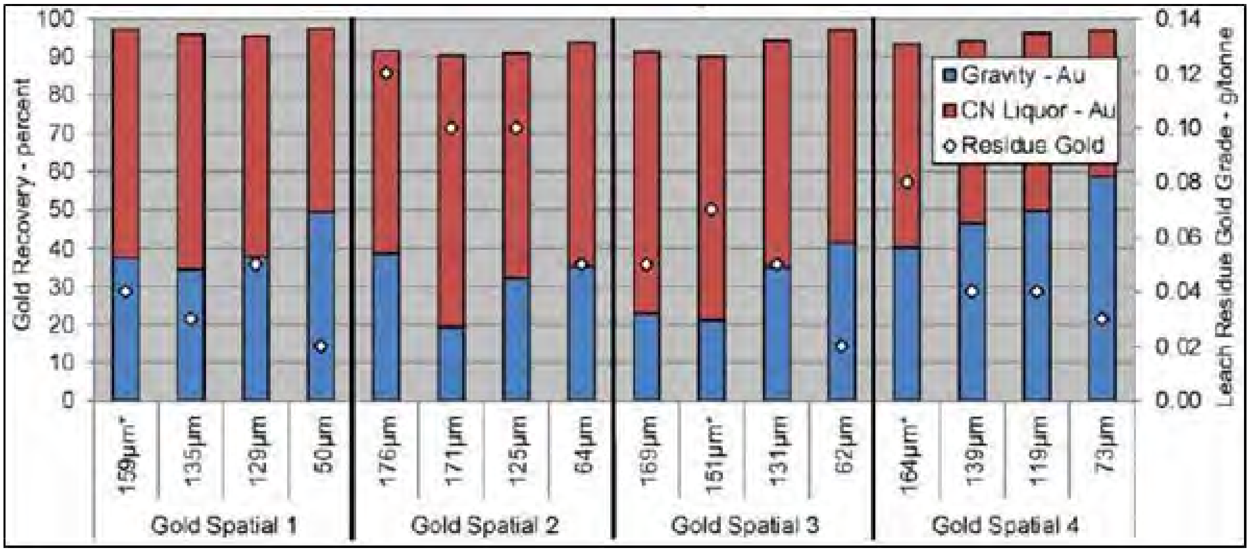
Source: ALS, 2015

Ag assays of products were near or below detection limits, limiting accuracy of results for Ag performance.
* Reagent consumptions are displayed in kilograms reagent per tonne leach feed.

Au extractions at the "optimized" conditions ranged from about 90% to 97% of the Au in the composites. Gravity concentration recovered about 20% to 47% of the Au, while cyanidation leaching of the gravity tails extracted a further 48% to 70% of the Au.

The highest overall Au extractions recorded were generally at the finest grind sizing tested, ranging from 94% to 98%. However, grinding costs would be significantly higher at these sizing's; cyanide and lime consumptions in the leach extractions were generally higher at the finer grind sizing as well.

Figure 13-19 presents a graphical representation of the various responses to the various leach conditions.



Source: ALS, January 2015

Figure 13-19: Gold spatial composite result summary

13.11.9.6 Diagnostic Leach Results

The cyanidation leach tailings from Tests 7, 8 and 9, underwent diagnostic leach tests to determine the association of the unrecovered Au from these tests. The cyanidation leach tests had been performed on gravity tailings from tests conducted at progressively finer sizings on the Gold Bulk Composite. A summary of the results is provided in Table 13-64.

Table 13-64: Diagnostic Leach Results

Gold Association	Test 7 159 µm P_{80}	Test 8 108 µm P_{80}	Test 9 56 µm P_{80}
Cyanidable Gold	2.7	36.2	41.4
Carbonate Locked Gold	56.5	24.5	13.9
Arsenical Mineral	7.7	13.5	15.4
Pyritic Sulphide Mineral	18.5	0.0	0.0
Silicate Encapsulated	14.5	25.9	29.3
Au In Leach Tail Percent	5.0	7.5	6.8

Source: ALS, 2015. Above values are displayed in percent.

More of the unrecovered Au was found to be cyanide extractable at grind sizings finer than 159 µm P_{80}. A slightly lower Au grade was also seen in the cyanidation residues from the tests at the finer grinds, this suggests that lower cyanidation leach Au losses at fine grinds may be possible.

13.11.10 ALS Saprolite and Transitional Mineralized Material Test Work

13.11.10.1 Saprolite Composite Testing

A total of 23 designated "Saprolite" composites were prepared from the Toroparu Deposit samples. Thirteen of the samples were designated Saprolite Volcanics (SV), these were sub composites SV1 though SV12 and SV Bulk Composite; ten composites were designated Saprolite Intrusives (SI), these were S11 though S19 and SI Bulk Composite.

An attrition scrubbing process was conducted prior to testing on all the Saprolite composites. This was performed for 30 minutes in a grinding mill without grinding media at 40% solids. The chemical analyses and metallurgical testing conducted on the Saprolite composites were conducted on the discharges from this process. These composites underwent various chemical analyses and testing involving gravity separation and cyanide leach extraction.

13.11.10.2 Chemical Content

The chemical content of the composites was analyzed through various methods. Table 13-65 displays a summary of the SV Au assays and Table 13-66 displays a summary of the SI Au assays. Screened Metallic Gold assays and size by size assays were completed on several select Saprolite composites.

Table 13-65: Saprolite Volcanic Gold Content Summary

Composite	Assay – g/t			
	Au Client	Au(M)	Au Recal s x s	Au Calculated (test)
SV 1	0.74	0.78	-	0.48
SV 2	0.58	-	-	0.62
SV 3	1.26	-	-	0.48
SV 4	1.17	-	-	0.49
SV 5	0.58	0.68	-	0.65
SV 6	1.02	1.24	-	0.55
SV 7	0.48	-	1.58	-
SV 8	0.46	-	0.71	-
SV 9	2.24	-	0.62	-
SV 10	0.94	-	0.96	-
SV 11	0.40	-	0.17	-
SV 12	0.92	-	1.00	-
SV Bulk	0.90	0.84	0.93	0.83

Source: ALS, 2015

Table 13-66: Saprolite Intrusive Gold Content Summary

Composite	Assay – g/t			
	Au Client	Au(M)	Au Recal sxs	Au Calculated (test)
SI 1	0.89	-	-	0.43
SI 2	0.67	-	-	0.45
SI 3	0.62	-	-	-
SI 4	1.28	-	-	0.83
SI 5	0.61	0.62	-	-
SI 6	1.77	-	-	1.30
SI 7	1.48	1.26	-	1.20
SI 8	0.48	-	0.45	-
SI 9	2.87	-	3.62	-
SI Bulk	1.06	0.75	1.02	1.14

Source: ALS, 2015

There was significant variation between the Company provided assays and recalculated Au assays from testing and size by size assay results. The screened metallic head assays, although not conducted on all composites, measured closer Au assays. This suggests a nugget effect occurring with the Au, the screened metallic Au assay method should be used when measuring Au content in these samples.

13.11.10.3 Saprolite Composite Test Results

The Saprolite Bulk Composites (SV Bulk. SI Bulk), along with several of the sub composites (SV1 through SV6, SI1 though SI4, SI5 and SI6) were tested through gravity concentration followed by cyanide leaching of the gravity tails. A summary of the testing is displayed in Table 13-67.

Table 13-67: Saprolite Test Conditions and Result Summary

Composite	Test# Grav/Leach	Feed Size Um ^80	Recovery %			Residue Au	Reagent Cons. kg/t *	
			Gravity	Cyanide	Overall	g/t	NaCN	Lime
SV Bulk	77/87	53	52.4	46.3	98.8	0.01	0.7	2.3
	114/127	42	29.5	69.6	99.1	0.01	0.7	2.4
SI Bulk	78/88	98	28.7	70.1	98.9	0.01	0.3	2.1
	115/128	66	51.1	47.9	99.0	0.02	0.2	2.0
SV1	102/100	286	5.4	87.6	93.0	0.03	0.4	2.9
SV2	103/116	54	14.6	82.6	97.2	0.01	0.3	3.9
SV3	104/117	45	9.9	86.3	96.1	0.01	1.1	3.4
SV4	104/118	36	24.0	73.6	97.6	0.01	0.2	2.5
SV5	106/119	60	21.9	76.3	98.2	0.01	0.3	2.5
SV6	107/120	56	15.0	80.5	95.5	0.02	1.1	0.9
SI1	108/121	60	16.8	79.9	96.7	0.01	0.2	0.7

Composite	Test# Grav/Leach	Feed Size Um ^80	Recovery %			Residue Au	Reagent Cons. kg/t *	
			Gravity	Cyanide	Overall	g/t	NaCN	Lime
SI2	109/122	50	11.1	85.7	96.8	0.01	0.2	1.0
SI3	110/123	65	36.5	60.6	97.2	0.01	0.3	2.5
SI4	111/124	52	37.7	60.7	98.5	0.01	0.1	1.2
S6	112/125	71	34.6	64.5	99.0	0.01	0.3	4.2
SI7	113/126	85	32.6	66.35	98.9	0.02	0.3	3.12

Source: ALS, 2015

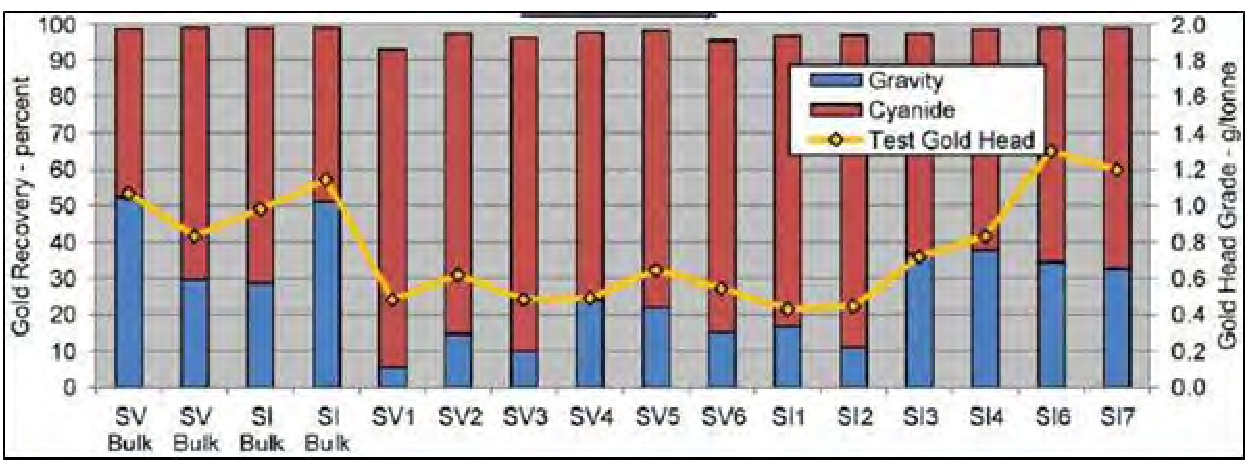
* Reagent consumption refers to kilograms reagent per tonne of leach feed

The feed for testing on the Saprolite composites was fed through an attrition scrubbing process and then screened. Coarser material (+150 µm) was reground and combined with the minus 150 µm material. The material underwent a Knelson Gravity concentration, followed by hand panning. The gravity tails were cyanide leached for 48 hours. The leach tests were conducted at a pH of 10.5 and pulp density of 40% solids. The initial cyanide concentration was 500 ppm sodium cyanide and maintained above 300 ppm throughout the 48 hours.

From testing of the two bulk composites, it could be seen that the gravity Au recovery varied dramatically. However, overall recoveries following the leach extractions were similar, as was the reagent consumption. Overall, about 99% of the Au was recovered from each of the bulk composites with between 29% and 52% of the Au being recovered in the gravity circuit and the remainder being recovered through the cyanide leach.

Overall Au recovery from the sub composites ranged from about 93% to 99% with gravity recovery ranging from 5% to about 38% of the feed Au and cyanide extractions accounting for a further 61% to 88% of the feed Au.

Gravity Au recovery trended generally with calculated gravity circuit Au feed grade. Figure 13-20 presents a graphical representation of the variability in the results.



Source: ALS January 2015. Displayed Au head grades are recalculated gravity circuit feeds.

Figure 13-20: Saprolite result summary

13.11.11 FLS Sedimentation Test Work

Samples of leach tails (including saprolite) and flotation tails were prepared for sedimentation test work. These samples were prepared according to the standard leach and flotation recipes determined in the metallurgical testing.

Flocculant screening tests, static settling tests and dynamic settling tests were conducted to obtain data for thickener design. No sample of final Cu concentrate was available for either settling tests or filtration tests. Of the flotation products for leaching (rougher scavenger concentrates and cleaner tails), only cleaner tails were available, and then only sufficient for static settling tests.

Flocculant screening indicated that Magnafloc 1011, a high molecular weight, anionic, medium charged flocculant produced the fastest settling rate and greatest supernatant clarity of all the samples tested. Settling tests results are found in Table 13-68.

Table 13-68: Summary of Settling Tests Design Data

Material Tested	Floc Consump. (g/mt)	Rise Rate (m/h)	Unit Area (m^2/tpd)	Yield Strength (Pa)	Slurry Density (% Solids)
Cleaner Tails (static test)	20	7	0.076	<30	52
Flotation Tails	25	5	0.054	<30	60
Leach Tails	60	4.5	0.059	<20	58

Source: FLS, 2014

13.11.12 ALS Carbon Characterization

A carbon triple contact carbon kinetic parameter determination was made on the Gold Bulk Composite and equilibrium characteristics were determined for the Gold Bulk Composite and the Gold Variability Composite. Results are summarized by Table 13-69 and Table 13-70.

Table 13-69: Carbon Kinetic Results

Leach Slurry Identity	Test	Fleming Adsorption Constants		Loaded Carbon Au Content (g/t)	
		k	n	Assayed	Calc'd
Gold Bulk Composite Test 79 Pulp	89	212	0.75	509	531

Source: ALS, 2015

Table 13-70: Carbon Equilibrium Data

Leach Slurry Identity	Test	Equilibrium Carbon Loading Au (g/t) @ Sol'n Concentration		
		0.50 ppm	0.20 ppm	0.10 ppm
Gold Bulk Composite	80/95	1,610	656	32
Gold Variability Composite	97/98	1,473	1,124	916
Gold Variability Composite	134/140	7,471	4,218	2,737

Source: ALS, 2015

13.11.13 Cyanide Detoxification

Continuous SO_2/air cyanide detoxification tests were undertaken on a master cyanidation tailings composite. The Master Composite was prepared by combining five cyanidation tailings samples and then contacting the slurry with 0 g/L activated carbon for 24 hours. The target WAD cyanide (CN_{WAD}) level for the treated effluent was <5 mg/L.

The initial level of CN_{WAD} in the leach effluent was estimated from the free cyanide titration together with analysis for Cu, Zn and Ni. The CN_{WAD} level was also determined directly by a colorimetric method using picric acid reagent. The two values agreed within the expected accuracy of the determinations.

An operating pH of 8.5 was initially chosen for the testwork as this represents a typical optimum level for the SO_2/air detoxification process. A residence time of nominally 90 minutes was chosen.

Both feed effluent and treated effluent solutions were analyzed by ALS Metallurgy for CN_{WAD} using a direct spectrophotometric determination with picric acid reagent. Cyanide determinations using this method have been identified in this report as CNP. Results are summarized by Table 13-71.

Cyanide speciation on the detoxification effluent was performed by the Chemistry Centre of Western Australia. The speciation is summarized by Table 13-72.

Table 13-71: Cyanide Detoxification Conditions and Results

Test ID	Test Conditions					Solution Assays	
	pH	Retention Time (minutes)	Reagents Used			Feed Effluent CNP (mg/L)	Treated Effluent CNP (mg/L)
			SO2 (g/g CN_{WAD})	CuS04.5H20 (mg/L)	Lime (g/g SO_2)		
Master Cyanidation Tailings Composite							
D1	8.69	84.88	5.00	82	0.53	221	1.06
D2	8.68	85.38	4.00	82	0.49	221	52.4
D3	8.73	57.49	5.00	82	0.44	221	0.24
D4	9.27	57.81	4.33	82	0.39	221	0.63

Source: Chemistry Centre of Western Australia,2014. CNp denoted determination by Picric Acid.

Table 13-72: Solution Analysis for Feed and Treated Effluent

Sample ID/Test No.	Cu (mg/L)	Fe (mg/L)	Ni (mg/L)	Zn (mg/L)	CNP (mg/L)	CN Total (mg/L)
Master Cyanidation Tailings Composite (Feed)	24.1	6.70	<0.05	0.36	221	240
Effluent D1	0.18	0.35	<0.05	<0.02	1.06	2.04
Effluent D2	13.3	<0.10	<0.05	0.02	52.4	52.5
Effluent D3	0.11	0.60	<0.05	<0.02	0.24	1.92
Effluent D4	0.07	3.05	<0.05	<0.02	0.63	9.16

Source: Chemistry Centre of Western Australia, 2014

The key findings of this test work were:

- The Toroparu Deposit mineralized material is amenable to the SO_2/air process with a residual CN_{WAD} level less than 5 mg/L able to be achieved;
- It was found that a Cu excess of 30 mg/L was necessary to achieve the target CN_{WAD} level; and

- When the SO_2:CN_{WAD} mass ratio was 4:1 and the pH was maintained at 8.5, the residual CN_{WAD} level was found to be high. However, when a pH of 9.0 was maintained at the same SO_2:CN mass ratio, the target CN_{WAD} level was able to be attained. This suggests some pH sensitivity and the opportunity to further optimize the process.

13.11.14 ALS Viscosity Testing

Viscosity results were undertaken on a wide range of composites including the saprolite composites. Various pulp densities were tested at various shear rates to reflect the range of shear rates as would be expected to be experience in a typical processing plant.

Many of the saprolite composites presented viscosities that were too high to measure, particularly at the higher pulp densities. As such, process design would need to consider the materials handling aspects associated with these types of feed sources. Results are presented in Table 13-73.

Table 13-73: Viscosity Results

Sample	Test	pH	Viscosity (cps) at Shear Rate (119.6 s-1)			Viscosity (cps) at Shear Rate (4.1 s-1)		
			Percent Solids			Percent Solids		
			60	50	40	60	50	40
Cu Bulk Comp Ro Tail	5	10.4	80	48	38	-	-	-
AuBulk Comp-CN Tail	7	8.6	75	53	38	-	-	-
AuBulk Comp-CN Tail	8	8.5	84	47	38	-	-	-
AuBulk Comp-CN Tail	9	8.8	139	54	35	-	-	-
SV1 Comp-CN Tail	100	11.0	*	820	121	*	12,070	1,315
SV2 Comp-CN Tail	116	10.5	*	202	78	*	2,999	620
SV3 Comp-CN Tail	117	10.5	*	140	56	*	-	-
SV4 Comp-CN Tail	118	10.8	*	363	64	*	2,753	-
SV5 Comp-CN Tail	119	10.9	*	222	68	*	1,793	455
SV6Comp-CN Tail	120	10.5	775	127	54	4,178	1,081	-
SI1 Comp-CN Tail	121	10.5		62	42	2596	-	-
SI2 Comp-CN Tail	122	10.5	345	68	39	2368	-	-
SI3 Comp-CN Tail	123	10.9	*	374	71	*	4,207	553
SI4 Comp-CN Tail	124	10.5	1256	117	44	8,795	760	-
SI6 Comp-CN Tail	125	10.5	*	438	91	*	5,200	964
SI7 Comp-CN Tail	126	10.5	*	237	53	*	1,840	-
SV Bulk Comp-CN Tail	127	11.0	*	416	70	*	2,892	494
SI Bulk Comp-CN Tail	128	10.5	*	218	59	13,880	2,072	-
Average			386	223	59	6,363	3,424	734
Minimum			75	47	35	2,368	760	455
Maximum			1,256	820	121	13,880	12,070	1,315

Source: ALS, Jan 14,2015. * Sample too viscous to measure. Viscosity too low to measure.

The spatial composite viscosity was determined as part of the ALS cyanide detoxification test work. The sample submitted showed low viscosity characteristics at a range of pulp densities and shear rates suggesting there should be no viscosity issues associated with processing these mineralized material types post detoxification. The results are summarized by Table 13-74.

Table 13-74: Slurry Viscosity Testing, post Cyanide Detoxification

Sample ID	% Solids (w/w)	Bohin Visco 88 Viscosity @ Shear Rate (sec-1) (cps)							
		4.2	7.4	13.1	21.9	38.9	67.4	119.2	209.5
Detoxification Slurry (ex Test D3)	60	Low	Low	Low	Low	44	51	65	97
	50	Low	Low	Low	Low	Low	21	37	61
	40	Low	Low	Low	Low	Low	16	25	47

Source: ALS, January 2015

13.11.15 ALS Variability Testing and Copper Solubility Test Work

Ten primarily Au composites designated SC Composites, and a second set of 15 Transition Composites were submitted to ALS Kamloops for variability testing. The two sample groups focused on different objectives, which can be summarized as follows:

- Assess the chemical content of the 10 SC Composites, along with the 15 Transition Composites;
- For the SC Composites, conduct gravity concentrations followed by cyanidation leach extractions on the gravity tailings; and
- For Transition Composites, perform a water soluble Cu extraction, followed by a 24-hour Chemical Content of the Composites.

The content of key elements of interest for the 10 SC Composites and 15 Transition Composites was assessed using standard analytical methods. The SC Composites were assayed in duplicate for Cu, Ag and Au. A summary of these average assays can be found in Table 13-75.

Table 13-75: SC Composite Head Assay Summary

Composite	Assay % or g tonne		
	Cu	Ag	Au
SC-3	0.04	<1	0.60
SC-4	0.08	<1	0.87
SC-5	0.04	<1	0.93
SC-6	0.04	1	0.60
SC-9	0.05	<1	0.38
SC-11	0.04	<1	0.64
SC-12	0.06	<1	0.40
SC-16	0.04	<1	0.54
SC-18	0.03	<1	0.89
SC-19	0.02	<1	1.00

Source: ALS, Jan 26, 2015. Copper assays are displayed in percent, Ag and Au assays are displayed in g/t.

The 15 Transition samples were assayed for Cu and Au. These assays are displayed in Table 13-76.

Table 13-76: Transition Composite Head Assays

Composite	Assay % or g/ tonne	
	Cu	Au
T_TRANS_1	0.05	0.39
T_TRANS_2	0.11	0.82
T_TRANS_3	0.03	0.34
T_TRANS_4	0.08	0.52
T_TRANS_5	0.17	1.98
T_TRANS_<S	0.36	0.79
T_TRANS_7	0.05	0.88
T_TRAN S_8	0.02	0.48
T_TRANS_9	0.07	0.79
T_TRANS_10	0.04	0.68
T_TRANS_11	0.16	0.36
T_TRANS_12	0.04	0.56
T_TRANS_13	0.06	1.19
T_TRANS_14	0.10	0.54
T TRANS 15	0.03	1.89

Source: ALS, January 2015. Copper assays are displayed in percent, Ag and Au assays are displayed in g/t.

Copper within the composites assayed between 0.02% and 0.36%. Within either sample group, the significance of this Cu on a cyanidation leach would depend on whether the Cu is present in a cyanide soluble mineral, which can increase cyanide consumption.

13.11.15.1 SC Composite Test Results

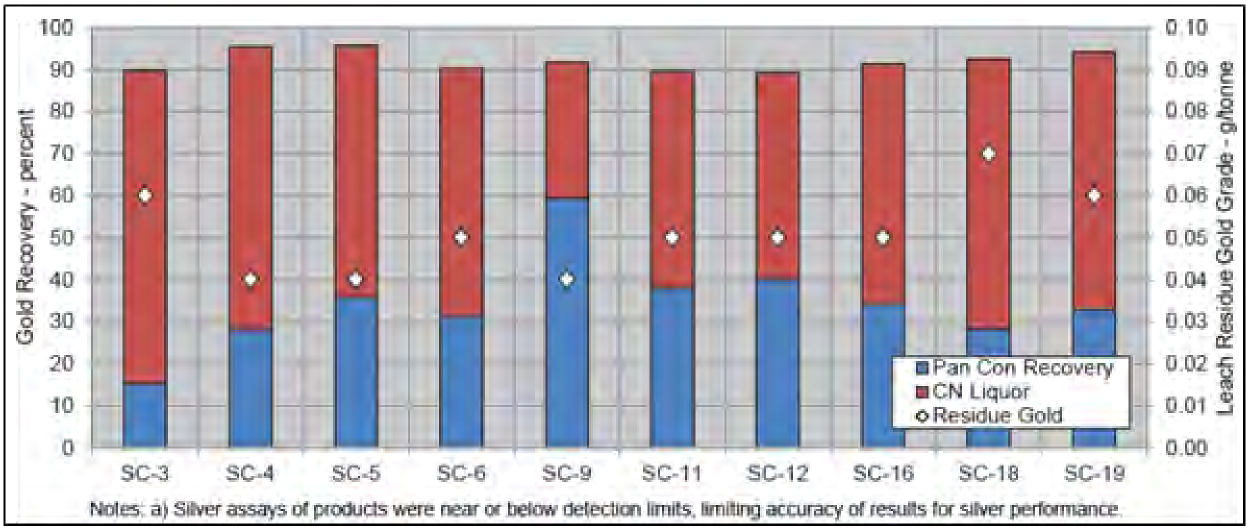
Table 13-77 displays a summary of the results from the metallurgical testing on the SC Composites while Figure 13-21 provides a graphical representation of the same data.

Table 13-77: SC Summary Leach Results and Test Conditions

Composite	Grind Size P_{80}	Au Extraction		Percent Overall	Residue Au g/t	Reagent Cons, kg/t Feed	
		Gravity	Leach			NaCN	Lime
SC-3	188	15.3	74.4	89.7	0.06	0.2	0.4
SC-4	113	28.2	67.2	95.4	0.04	0.4	0.5
SC-5	150	35.9	59.8	95.7	0.04	0.2	0.4
SC-6	141	31.2	59.1	90.3	0.05	0.3	0.4
SC-9	109	59.5	32.2	91.7	0.04	0.2	0.4
SC-11	160	38.1	51.5	89.6	0.05	0.2	0.4
SC-12	127	40.1	49.2	89.3	0.05	0.3	0.4

Composite	Grind Size P_{80}	Au Extraction		Percent Overall	Residue Au g/t	Reagent Cons, kg/t Feed	
		Gravity	Leach			NaCN	Lime
SC-16	130	34.3	57.0	91.3	0.05	0.2	0.4
SC-18	98	28.3	64.2	92.5	0.07	0.2	0.5
SC-19	107	32.9	61.3	94.2	0.06	0.2	0.5

Source: ALS, January 2015



Source: ALS, January 2015

Figure 13-21: Graphical representation of the SC leach results

The flowsheet involved primary grinding the composites, targeting 150 µm P_{80} discharge sizing followed by Knelson Gravity separation with hand panning of the Knelson concentrate to produce a gravity concentrate. Gravity tailings were then subjected to a cyanidation leach test. The cyanidation

leach was conducted over 48 hours with an initial 500 ppm sodium cyanide concentration that was maintained at over 300 ppm sodium cyanide for the duration of the test. Cyanide liquors were sampled at 2, 4, 6, 24 and 48 hours during the tests. The leach tests were conducted with a pH target of 10.5 with oxygen sparging at each sampling interval.

Gravity recoveries across the 10 SC Composites ranged from about 15% to 60% of the feed Au, and the subsequent leaching of the gravity tailings brought the overall recovery of Au to between 89% and 96%. The Au grade of the leach residues ranged from about 0.04 g/t to 0.07 g/t. The cyanidation leach tests consumed between 0.2 kg and 0.4 kg of sodium cyanide per tonne of leach feed and between 0.4 kg and 0.5 kg of lime per tonne feed. The highest sodium cyanide consumption was measured for the SC-4 Composite, which also measured the highest feed Cu grade.

Overall Ag recoveries ranged from 70% to 89%; however, due to the low feed Ag grade and tailings measuring at or below detection limits, the values for Ag recovery may not be accurate.

13.11.15.2 Transition Composite Test Results

Table 13-78 displays a summary of the testing conducted on the 15 Transition Composites. The test procedure performed on these composites consisted of pulverizing a sample of each composite, followed by a 2-hour bottle roll agitation of the slurry with no reagents. The liquor was sampled and analyzed for Cu to determine water soluble Cu content. Following a 2-hour agitation, a cyanidation leach of the slurry was conducted over 24 hours. The slurry was raised to a pH of 10.5 with an initial sodium cyanide concentration of 500 ppm. The sodium cyanide concentration was maintained above 300 ppm over the course of the test and the slurry was sparged with oxygen at the monitoring intervals.

Table 13-78: Transitional Summary Leach Results and Test Conditions

Composite	Grind Size µm P_{80}	Natural pH	Water Soluble Cu %	Extraction %		Residue Grade g/t		Reagent Cons, kg/t Feed	
				Au	Ag	Au	Ag	NaCN	Lime
T -TRANS -1	27	7.9	0.0	83.7	63.5	0.04	0.1	0.2	3.2
T_TRANS_2	20	8.1	0.0	94.0	32.2	0.03	0.4	0.3	4.0
T_TRANS_3	21	8.1	0.0	95.8	65.3	0.01	0.2	0.2	3.1
T_TRANS_4	24	8.3	0.0	98.1	87.2	0.01	0.1	0.5	4.1
T_TRANS_5	28	8.6	0.0	11.5	4.4	2.25	1.8	1.8	3.2
T_TRANS_6	22	8.0	0.0	33.7	1.5	0.54	6.1	1.9	3.3
T_TRANS_7	28	8.1	0.0	96.0	81.8	0.04	0.3	0.7	3.5
T_TRANS_8	30	8.8	1.0	80.9	91.6	0.05	0.1	0.6	4.4
T_TRANS_9	40	7.9	0.0	53.2	93.9	0.23	0.4	0.8	3.8
T_TRANS_10	25	8.2	0.1	91.9	87.9	0.03	0.2	0.8	3.0
T_TRANS_11	22	8.2	0.0	24.0	14.5	0.31	3.9	1.6	3.2
T_TRANS_12	20	7.8	0.0	95.4	90.8	0.04	0.1	0.4	3.7
T_TRANS_13	26	8.3	0.0	96.9	80.5	0.03	1.1	1.0	3.7
T_TRANS_14	27	8.5	0.0	80.3	68.2	0.09	0.5	1.1	3.9
T_TRANS_15	34	7.7	0.0	96.3	62.5	0.04	0.1	0.2	5.5

Source: ALS, January 2015

Results from the 2-hour pre-cyanide agitation indicated that very little to no Cu from the Transition Composites was water soluble. The highest percentage of solubilized Cu measured only 1% of the Cu in the feed for the T_TRANS_8 Composite. The natural pH of the samples ranged from 7.7 to 8.6 suggesting the pulps were not acid and this correlates with the results observed as acid pulps would have suggested mobilization of Cu was probable.

Au and Ag extractions over the 24-hour cyanidation leach varied considerably, ranging from about 12% to 98% for Au and between 2% and 94% for Ag extraction. Composites with lower extractions, such as T_TRANS_5, T_TRANS_6 and T_TRANS_11 generally consumed more cyanide suggesting the presence of other cyanide consuming mineral species inclusive of Cu.

13.12 Metallurgical Test Work 2018 – 2020 (Sona Hill)

13.12.1 Base Metallurgical Laboratories Program BL231 (April 2019)

Test work performed at Base Metallurgical Laboratories (BML) was to assess the metallurgical performance of samples, provide data from process optimization and variability testing, and generate metallurgical data for the Sona Hill Deposit. Test work consisted of process development on three master lithological composites, representing the average resource as well as using the developed process to evaluate a set of Variability samples. Approximately 889 kg of sample, as quarter drill core, were received at BML between August 2017 and April 2018.

Three master composites, SAP-MC (Saprolite), GRDT-MC (Granodiorite) and GRDT-QZ (Granodiorite high quartz) were made to represent the three lithologies identified at Sona Hill.

13.12.1.1 Chemical Composition

Preliminary head analyses for the three master composites are provided in Table 13-79.

Table 13-79: Head Assays

Composite	Assay % (Cu, Fe, S) or g/t (Au, Ag, Te)					
	Au	Ag	Cu	Fe	S	Te
SAP-MC	1.59	6	0.010	10.2	0.03	10
SAP-MC Screen Metallics	1.49					
GRDT-MC	2.81	6	0.009	6.5	0.72	6
GRDT-MC Screen Metallics	3.01					
GRDT-QZ	3.70	10	0.008	5.4	0.86	11
GRDT-QZ Screen Metallics	3.56					

Source: BML, 2019

The SAP-MC, GRDT-MC and GRDT-QZ samples contained 1.5, 3.0 and 3.6 g/t of Au, respectively, via screen metallic fire assay. Tellurium was added to elements of analysis due to the Au associated with tellurium having a negative affecting on Au leaching.

13.12.1.2 Comminution Testing

BML conducted Bond ball mill work index (BWi), Bond rod mill work index (RMi), Abrasion index (Ai) and a Levin fine grinding test on select composites to determine comminution parameters for the Sona Hill mineralized material. A summary of the results is presented in Table 13-80.

Table 13-80: Comminution Results

Composite	kWh/tonne		Ai	Levin Test P_{80} µm at Power Input				
	WiBM	WiRM		Feed µm	5 kWh/t	10 kWh/t	15 kWh/t	30 kWh/t
SAP-MC	8.6		0.011	360	88	51	38	31
GRDT-QZ	12.3	14.1	0.186					

Source: BML, 2019

The BWi for the SAP-MC sample was determined to be 8.6 kWh/tonne, using a closing screen size of 150 µm, indicating this mineralization to be very soft. The Ai of this sample was measured to be 0.011, indicating the sample is not abrasive.

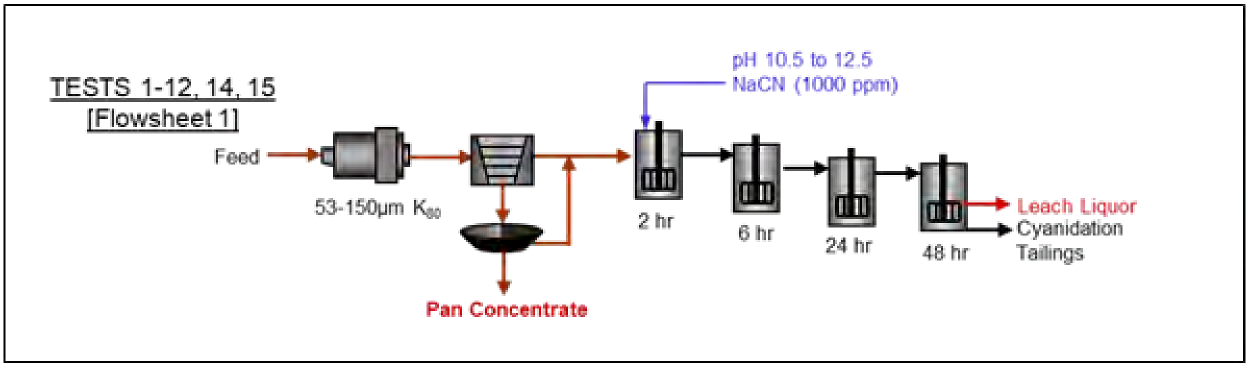
The BWi for the GRDT-QZ sample was determined to be 12.3 kWh/tonne at a closing screen size of 150 µm. The RMi for this sample was measured to be 14.1 kWh/tonne, which classifies the sample as moderately hard from a rod milling perspective. The Ai of this sample was measured to be 0.186, indicating the sample is mildly abrasive.

The Levin test conducted on the SAP-MC sample provided an estimate particle size based on grinding energy and a feed particle size of 360 µm.

13.12.1.3 Flowsheet Development

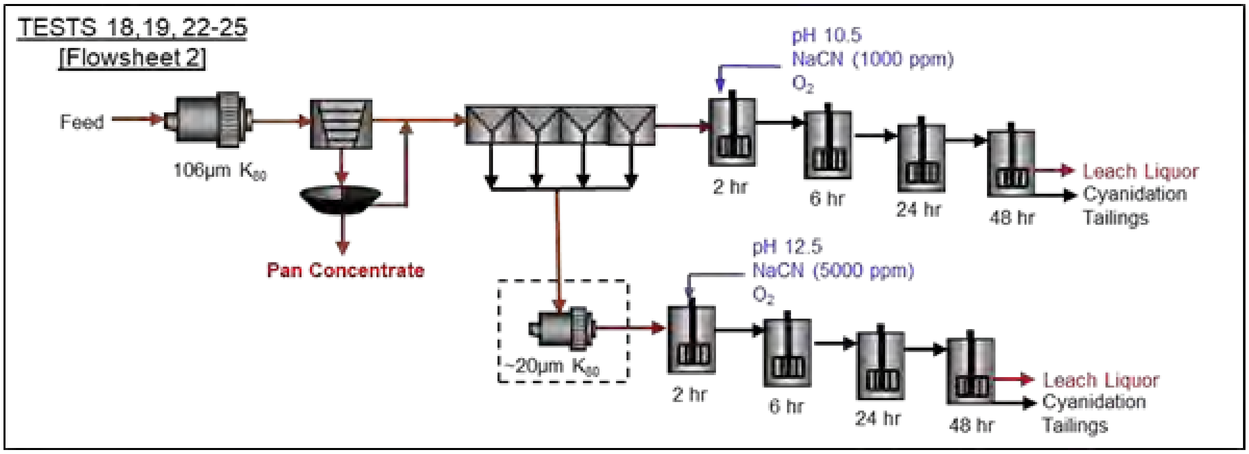
Three flowsheet configurations were tested in an attempt to maximize Au extraction. All flowsheets included a primary grinding stage, followed by Knelson gravity concentration, followed by cyanide leaching of the gravity tailings. The flowsheets evaluated implementing a rougher flotation stage prior to leaching the gravity tailings, as well as cyanide leaching of the rougher concentrate with and without a regrind stage. The test work further investigated the effect of cyanide leaching the gravity concentrate, with and without a regrind stage.

The three flowsheets are presented in Figure 13-22 to Figure 13-24.



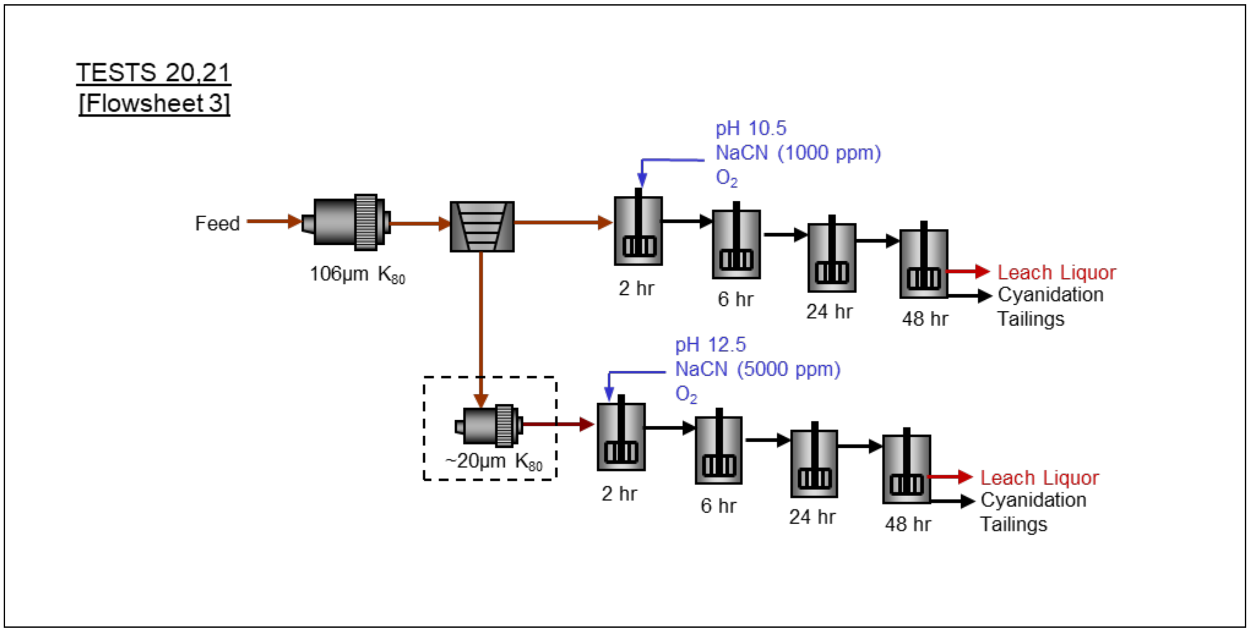
Source: BML, 2019

Figure 13-22: Standard flowsheet



Source: BML, 2019

Figure 13-23: Standard flowsheet with flotation and regrind



Source: BML, 2019

Figure 13-24: Standard flowsheet with regrind of the gravity concentrate

Test results indicated that Au from the SAP-MC sample presents high extractions of 94% to 98% when applying a Gravity Leach flowsheet. Au from the GRDT-MC presented extractions of 81% and 85% while the GRDT-QZ sample extractions were between 74% and 85% via the same flowsheet. Finer primary grinding of the samples resulted in a minor improvement in leach kinetics and Au extraction for these samples.

At a primary grind size of 53 µm P_{80}, testwork demonstrated that at pH of 12.5 a significant increase to Au extraction and leach kinetics of the gravity tailings occurred.

Implementing a rougher flotation stage after the gravity concentration, and regrinding the rougher concentrate prior to leaching, resulted in a significant increase in overall Au recovery; up to 96% for GRDT-MC and up to 97% for GRDT-QZ.

13.12.1.4 Multi-Stage Sequential Diagnostic Leach

Leach residue from Tests 8 and 12 were subjected to a multi-stage diagnostic leach test to determine the Au deportment in the leach residues. The diagnostic results indicate that the majority of the Au may be associated with arsenical minerals or minerals that are attacked by nitric acid (Table 13-81). However, it is most likely the association with tellurides is responsible for the bulk of the residual gold.

Table 13-81: Diagnostic Leach Results

Stage	Au g/t per stage	
	GRDT-MC T08-Residue	GRDT-QZ T12-Residue
Cyanidable Au	0.14	0.26
Carbonate Locked Au	0.04	0.03
Arsenical Mineral (Arsenopyrite)	0.23	0.45
Pyritic Sulphide Mineral	0.01	0.01
Silicate (Gangue) Encapsulated	0.03	0.05
Total (recalculated) Au Grade	**0.46**	**0.80**
Measured Au Grade	0.42	0.73
	Au Distribution %	
Cyanidable Au	30.4	32.4
Carbonate Locked Au	9.4	3.8
Arsenical Mineral (Arsenopyrite)	51.0	55.8
Pyritic Sulphide Mineral	2.7	1.7
Silicate (Gangue) Encapsulated	6.5	6.2
Total	**100.0**	**100.0**

Source: BML, 2019

13.12.1.5 Scrubbing Test

A single scrubbing test was conducted on the SAP-MC sample. The products from the scrubbing test were dried, weighed, and assayed to determine the mass and Au deportment. About 21.5% of the Au, assaying 0.73 g/t, was contained in the -150 µm fraction size, which contained 58% of the mass. Results for the scrubbing test are presented in Table 13-82.

Table 13-82: SAP-MC Scrubbing Test Results

Sieve Size Microns	Feed Au			Scrubbing Product Au		
	Mass %	Assay g/t	Distribution %	Mass %	Assay g/t	Distribution %
3,350	0.2	3.16	0.4	0.1	6.33	0.3
2000	5.7	3.16	11.0	4.4	6.33	14.3
425	25.6	3.44	53.7	23.4	4.27	50.8
150	13.7	1.48	12.3	13.7	1.89	13.2
-150	54.8	0.68	22.6	58.3	0.73	21.5
Total	**100**	**1.64**	**100**	**100**	**1.97**	**100**

Source: BML, 2019

The scrubbing test demonstrated that this procedure would not be successful at generating a product stream that could be determined as waste.

13.12.1.6 Trace Mineral Search

A trace mineral search (TMS) was conducted on the rougher concentrates from Tests 18 and 19 and revealed that Au in the rougher concentrate for GRDT-MC occurred mainly as minerals containing tellurium (Calaverite, sylvanite, petzite) as well as locked binaries with pyrite and non sulphide gangue (Table 13-83). Approximately 8% of this Au was liberated. Au liberation in the GRDT-QZ rougher concentrate was much lower at 10.2%. Au liberation results are presented in Table 13-84. Over half the Au in the rougher composites for both samples, by mass, occurred as inclusions in larger particles, indicating that further regrinding would be necessary to expose the Au surface area for cyanide leaching. Extended leach times may also be warranted as minerals containing tellurium are known to be slower leaching.

Table 13-83: Gold Deportment by Mineral Species

Au Bearing Minerals	Mineral Distribution % Mass	
	Test 18A Rougher Concentrate	Test 19 A Rougher Concentrate
Native Gold	15.7	13.7
Electrum	7.05	5.84
Calaverite /Sylvanite	50.4	54.2
Petzite	26.5	26.2
Uytenbogaardtite	0.34	0.02
Total	**100**	**100**

Source: BML, 2019

Table 13-84: Gold Liberation by Association

Mineral Status	Mineral Distribution % Mass	
	Test 18A Rougher Concentrate	Test 19 A Rougher Concentrate
Liberated	27.8	10.2
Au Ag Binary	0.30	0.005
Au Py Binary	48.8	77.2
Au Os Binary	0.04	0.02
Au FeOx Binary	2.22	0.23
Au Cb Binary	2.80	0.88
Au Gn Binary	12.8	0.85
Au Multiphase	5.24	10.5
Total	**100**	**100**

Source: BML, 2019

13.12.1.7 Variability Testing

Six samples from GRDT-MC zone and six samples from the GRDT-QZ were used for variability metallurgical testing. The samples were selected to test a range of Au and tellurium grades as Au associated with tellurium can result in low extractions. The Au and tellurium grades tested are displayed in Table 13-85 for the 12 samples.

Table 13-85: Gold Liberation by Association

Sample	Assay g/t	
	Au	Te
S-09	2.65	1.20
S-20	11.0	17.6
S-24	1.00	4.40
S-36	1.38	0.10
S-43	1.70	2.10
S-46	1.69	4.80
S-59	2.53	16.7
S-61	1.01	1.30
S-68	1.89	5.60
S-77	2.79	25.1
S-79	1.35	0.50
S-84	3.72	76.7

Source: BML, 2019

Gravity leach tests on the Variability Composites, at a primary grind size of 53 µm P_{80}, demonstrated that leach kinetics improved, and overall Au extraction increased at a pH of 12.0 when compared to sample tested at a pH of 10. Table 13-86 summarizes the variability tests.

Table 13-86: Variability Samples Summary

SampleID	Test	pH	Au Extraction % Cumulative					Consumption kg/t		Head Assayg/t	
			0	2	6	24	48	NaCN	Lime	Au	Te
S-09	26	10	5.8	58.5	78.9	86.9	89.5	1.07	0.28	2.65	1.2
	27	12	8.1	44.6	78.2	90.0	91.5	0.68	2.66	2.65	1.2
S-20	28	10	18.4	32.4	48.2	66.9	74.3	0.64	0.21	11.0	17.6
	29	12	20.0	44.8	65.4	80.4	86.0	0.50	2.06	11.0	17.6
S-24	30	10	18.6	47.5	63.5	74.1	80.4	0.84	0.22	1.00	4.4
	31	12	6.5	44.0	69.0	86.6	88.9	0.61	1.80	1.00	4.4
S-36	32	10	16.5	43.5	65.2	78.6	85.1	1.15	0.17	1.38	0.1
	33	12	23.6	59.0	72.4	93.5	93.3	0.45	1.21	1.38	0.1
S-43	34	10	13.2	40.7	61.0	79.9	84.9	0.75	0.18	1.70	2.1
	35	12	11.4	54.2	69.9	81.6	87.5	0.50	2.09	1.70	2.1
S-46	36	10	6.4	40.6	62.4	76.0	84.6	0.96	0.15	1.69	4.8
	37	12	12.4	57.4	69.9	82.6	90.3	0.28	2.04	1.69	4.8
S-59	38	10	12.4	45.5	67.1	78.7	83.5	1.15	0.17	2.53	16.7
	39	12	22.3	68.6	78.8	84.8	86.2	0.52	1.98	2.53	16.7

SampleID	Test	pH	Au Extraction % Cumulative					Consumption kg/t		Head Assayg/t	
			0	2	6	24	48	NaCN	Lime	Au	Te
	50	12*	26.4	54.4	67.4	75.6	85.4	0.84	4.00	2.53	16.7
S-61	40	10	10.7	27.4	60.9	69.1	80.5	1.17	0.15	1.01	1.3
	41	12	10.5	48.1	68.9	83.8	84.7	0.55	1.40	1.01	1.3
S-68	42	10	13.6	22.6	45.3	65.7	78.7	1.24	0.15	1.89	5.6
	43	12	16.2	53.0	65.6	86.1	91.2	0.39	2.36	1.89	5.6
S-77	44	10	12.5	46.5	63.0	75.2	82.0	1.10	0.16	2.79	25.1
	45	12	8.5	58.5	70.6	82.9	85.6	0.61	2.67	2.79	25.1
S-79	46	10	11.9	64.8	85.3	88.5	91.7	0.90	1.51	1.35	0.5
	47	12	13.7	68.2	82.0	85.1	88.2	0.27	8.58	1.35	0.5
S-84	48	10	17.6	35.7	48.4	66.9	76.5	1.22	0.20	3.72	76.7
	49	12	16.3	48.5	61.9	83.4	91.6	0.48	1.75	3.72	76.7

Source: BML, 2019

13.12.2 Base Met Labs Program BL473 (December 2019)

13.12.2.1 Condition Optimization

A follow up test work program (BL473) was conducted by BML to explore optimization conditions given the observations regarding fine grind and high pH leaching outcomes for high Te samples. The work reported in December of 2019.

The two master composites GRDT-MC and GRDT-QZ were used for the work given these composites provided adequate sample mass on which to conduct a matrix of tests.

The test work program included:

- Testing of primary grind size sensitivity.
- NaCN concentration influence.
- Pre-aeration and oxygen sparging.
- Use of $PbNO_3$.
- Use of NaOH for pH control.
- Leaching undertaken at a target pH of 12.5 with some comparative tests undertaken at pH 10.
- The flowsheet remained as grind, gravity and leaching of the gravity tails with or without pre-aeration.

  The results of the program are summarized by

Table 13-87.

Table 13-87: Optimization Conditions Test Work Summary

Composite	Test	PG	pH	NaCN	Other Conditions	Au Rec 72 Hours*	CN Tail g/tonne	Consumption kg/tonne	
		pm K_{80}		g/tonne		%	Au	NaCN	Lime
GRDT-MC	BL231-05	150	10	1,000	-	80.8	0.46	0.34	1.09
	BL231-06	106	10	1,000	-	83.8	0.41	0.37	1.08
	BL231-07	75	10	1,000	-	85.2	0.38	0.57	1.14
	BL231-08	53	10	1,000	-	82.7	0.42	0.73	1.04
	BL231-14	53	12.5	1,000	-	92.6	0.29	0.30	6.12
	1	106	12.5	1,000	-	85.0	0.36	0.49	20.1
	2	125	12.5	1,000	-	83.9	0.48	0.43	21.9
	3	125	12.5	1,000	CIL	89.9	0.35	0.55	5.1
	4	125	12.5	1,000	$PbN0_3$	90.3	0.35	0.39	5.2
	5	125	12.5	1,000	NaOH	90.1	0.31	0.28	4.3
	6	125	12.5	500	-	86.7	0.36	0.28	6.1
	7	125	12.5	250	-	84.8	0.44	0.16	5.7
	8	125	12.5	1,000	pre-air	83.4	0.45	0.39	12.4
	9	125	12.5	500	pre-air	85.9	0.38	0.28	10.7
	15	125	12.5	1,000	pre-air more DO	82.6	0.47	0.34	7.8
	16	125	12.5	500	pre-air more DO	85.8	0.35	0.20	7.9
GRDT-QZ	BL231-09	150	10	1,000	-	74.1	0.81	0.37	0.6
	BL231-10	106	10	1,000	-	76.1	0.68	0.55	0.6
	BL231-11	75	10	1,000	-	80.7	0.54	0.54	0.6
	BL231-12	53	10	1,000	-	78.2	0.73	0.75	0.7
	BL231-15	53	12.5	1,000	-	84.7	0.40	0.23	8.0
	10	106	12.5	1,000	-	84.6	0.49	0.45	5.1
	11	125	12.5	1,000	-	81.6	0.63	0.43	4.8
	12	125	12.5	1,000	$PbN0_3$	81.7	0.63	0.37	4.9
	13	125	12.5	1,000	pre-air	81.7	0.61	0.37	10.6
	14	53	10.0	1,000	-	81.5	0.52	0.61	0.2
	17	125	12.5	1,000	$PbN0_3$	75.4	0.74	0.31	7.4
	18	125	12.5	1,000	pre-air	78.4	0.67	0.33	7.7

Source: BML, 2019

The results showed some grind sensitivity to leach extraction at a pH of 10. However, the benefit was observed to be limited and potentially not economic to chase finer and finer grinds.

Coarser grind sizes in the 106 to 125 µm range at elevated pH of 12.5 typically presented higher extractions than those achieved at finer grinds and low pH.

Two tests conducted at 53 µm and high pH provided the highest extractions/lowest leach residue grade.

Lead nitrate appeared to provide a benefit for the GRDT-MC sample but not so for the GRDT-QZ sample. A single caustic soda test also showed some promise as did a single CIL test on the GRDT-MC sample. The lead nitrate, caustic soda and pre-aeration tests all presented a reduced cyanide consumption adding benefit over and above the apparent extraction benefits.

The program highlighted a number of opportunities with regard to optimizing leach extraction and also suggested that it may be possible to blend the Sona Hill Deposit material with the Toroparu Deposit materials at a coarser grind and still achieve adequate leach extractions if the leach conditions were manipulated to suit the Sona Hill Deposit material.

Of note, the quicklime consumptions were very erratic for the high pH tests. Two GRDT-MC tests presenting consumption values of around 20 kg/t which do not align with the remaining tests. These consumptions may have been a result of pH measurement issues. It is also noted that the pre-aeration tests also have higher quicklime consumption for both master composites.

13.12.3 Base Met Labs Program BL523 (February 2020)

13.12.3.1 Full-scale Processing Considerations

As part of the project development strategy, it was proposed the Sona Hill Deposit material would be processed by blending with other the Toroparu Deposit material including saprolites. Consequently, it was decided a test work program was required to help understand how the materials would perform in various blends. The test work program to also determine the performance of a number of Sona Hill Deposit variability samples. Base Met Labs conducted a program in late 2019 and reported the outcomes in early 2020.

13.12.3.2 Program Detail and Results

The samples used for the program included various master composites, bulk composites and variability composites generated from earlier programs. In addition, some 12 existing variability samples were retrieved for testing. The samples used and their assays are summarized by Table 13-88.

Table 13-88: Program BL523 Samples and Head Assays

Sample ID	Au g/t	Ag g/t	Cu %	Fe %	S %
SAP-MC	1.59	6.0	0.010	10.2	0.03
GRDT-MC 2	1.62	1.9	0.012	4.90	0.91
TX-MC	5.48	4.1	0.008	5.44	0.53
FR-MC	1.99	1.3	0.010	4.72	0.84
Bulk Comp	2.50	2.4	0.009	6.13	0.62
Gold Bulk Comp	0.86	1.1	0.039	3.20	0.12
Gold Var Comp	2.12	0.4	0.081	3.34	0.14
S-05	0.89	-	-	-	-
S-16	14.4	-	-	-	-
S-21	1.55	-	-	-	-
S-30	0.75	-	-	-	-
S-36	1.38	-	-	-	-
S-39	1.62	-	-	-	-
S-43	1.70	-	-	-	-
S-48	0.84	-	-	-	-
S-52	2.79	-	-	-	-
S-61	1.01	-	-	-	-
S-69	4.12	-	-	-	-
S-83	1.31	-	-	-	-

Source: BML, 2020

The program evaluated:

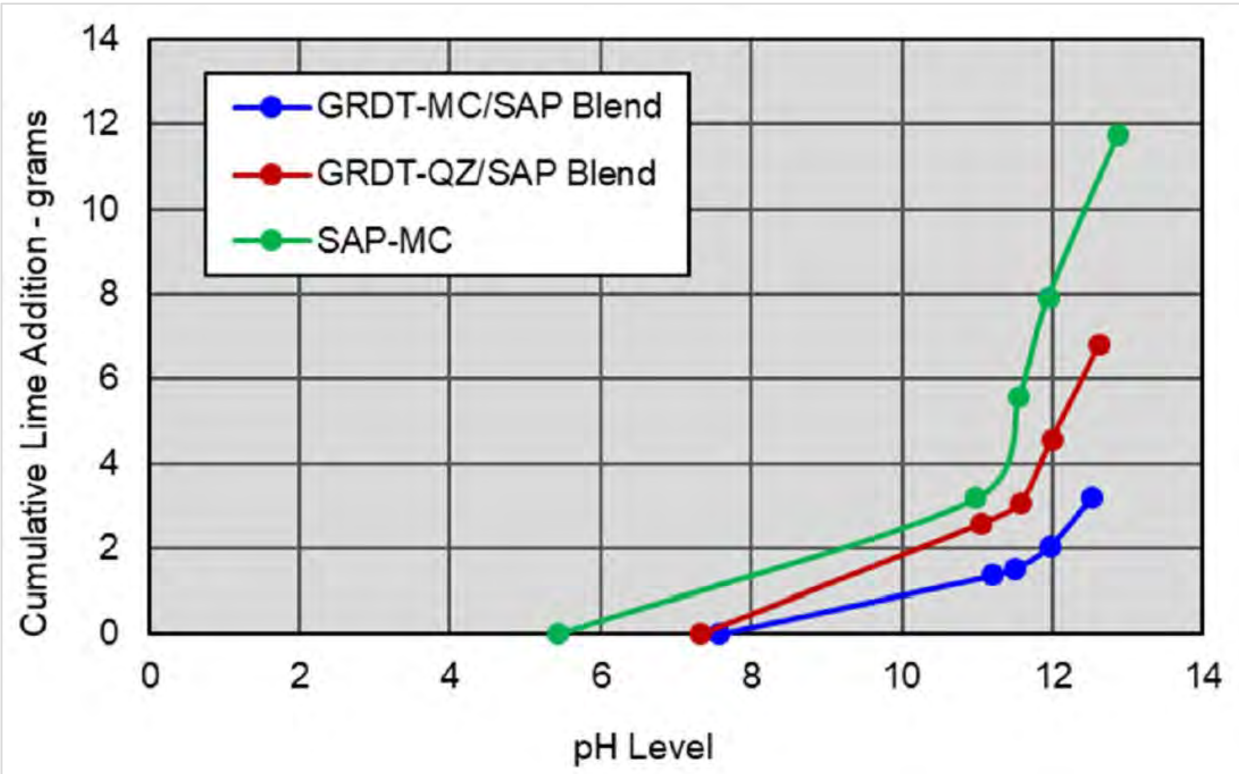
- Quicklime (lime) consumption for various blends.
- Gravity leach testing of various blends.
- Carbon characterization test work at high pH.
- Cyanide detoxification test work of blended material.
- Flocculant screening tests.
- Static and Dynamic settling tests.
- Yield stress and sheared viscosity testing.

The quicklime consumption tests were undertaken at 100% saprolite master composite (SAP-MC) and a blend of 0.38 to 1 kg SAP-MC to either the GRDT-MC or GRDT-QZ master composites. The ratio selected based on proposed mine plans of the time. Figure 13-25 presents the results.

The quicklime demand for the SAP-MC is around 12 kg to achieve a pH of 12.5. This suggests the quicklime consumption for GRDT-QZ is 4 kg/t to achieve a pH of 12.5 when mixed with SAP-MC. This is in the lower consumption range for GRDT-QZ when compared to

Table 13-87 values.

However, the consumption for GRDT-MC would need to be nominally *negative* 1.5 kg/t for the consumption to align by ratio. The work suggests that there is some synergistic benefit in blending the material types with the potential that the lower pH saprolite is being partially neutralized by the other master composites. Effectively, the suggests quicklime consumption estimates based on blended demands will overstate the quicklime consumption in practice.



Source: BML, 2020

Figure 13-25: Quicklime consumption of blends

A series of gravity leach tests were run on a mix of both Sona Hill variability samples as had been tested in earlier programs as well as samples that had been prepared for earlier programs and not tested. In addition, blends of the various master composites, bulk composites, and variability samples with saprolite (SAP-MC) were subjected to gravity leach testing to evaluate if test results of blends were as anticipated.

The leach time was extended out to 96 hours. This was due to the observation from earlier work that while the bulk of the leaching was complete at 24 hours, the leach tests showed a long tail and some samples presented much higher extractions at extended leach durations. Notably the higher Te samples. Results of the work are summarized by Table 13-89 where the total recovery includes the gravity component.

Table 13-89: Program BL523 Gravity Leach Result Summary

Sample ID	Test	Leach Res Sizing pm K_{80}	NaCN ppm	Au Rec 96 hours percent	CN Tails g/tonne Au	Consumption kg/tonne NaCN	Consumption kg/tonne Lime
S05	1	72	1,000	82.4	0.14	0.45	5.02

Sample ID	Test	Leach Res Sizing pm K_{80}	NaCN ppm	Au Rec 96 hours percent	CN Tails g/tonne Au	Consumption kg/tonne	
						NaCN	Lime
S16	2	54	1,000	98.3	0.24	0.38	7.93
S16	18	53	500	97.0	0.47	0.18	6.96
S16/SAP-MC Blend	14	61	1,000	95.5	0.45	0.39	5.98
S21	3	108	1,000	70.7	0.46	0.34	4.19
S30	4	44	1,000	94.4	0.05	0.31	5.39
S36	5	77	1,000	85.5	0.13	0.34	4.42
S39	6	59	1,000	87.8	0.19	0.36	4.43
S39	19	61	500	91.9	0.17	0.18	4.07
S43	7	80	1,000	85.7	0.26	0.39	4.22
S43/SAP-MC Blend	15	83	1,000	87.5	0.23	0.28	4.40
S48	8	94	1,000	87.1	0.12	0.28	4.03
S52	9	78	1,000	86.7	0.34	0.52	4.71
S61	10	90	1,000	87.2	0.11	0.30	3.91
S61/SAP-MC Blend	16	92	1,000	90.3	0.12	0.27	4.34
S69	11	71	1,000	76.5	0.84	0.36	4.00
S83	12	66	1,000	86.0	0.17	0.37	4.18
S83	20	62	500	89.4	0.15	0.10	4.05
S83/SAP-MC Blend	17	68	1,000	89.5	0.16	0.30	4.56
SAP-MC	13	79	1,000	95.7	0.07	0.41	7.61
SAP-MC	21	76	500	95.2	0.08	0.19	4.45
GRDT-MC2	22	73	500	87.2	0.23	0.30	5.56
GRDT-MC2/SAP-MC Blend	25	71	500	89.6	0.20	0.39	6.04
TX-MC	23	50	500	94.6	0.22	0.46	6.57
FR-MC	24	71	500	84.8	0.31	0.51	6.31
FR-MC/SAP-MC Blend	26	72	500	87.3	0.24	0.39	6.51
Bulk Comp	27	57	500	91.0	0.16	0.39	8.99
Gold Bulk Comp	29	150	500	93.6	0.04	0.36	0.60
Gold Bulk/SAP-MC	31	154	500	93.6	0.07	0.33	0.68
Gold Var Comp	30	150	500	91.7	0.06	0.64	0.28
Gold Var/SAP-MC	32	146	500	78.7	0.21	0.41	0.56

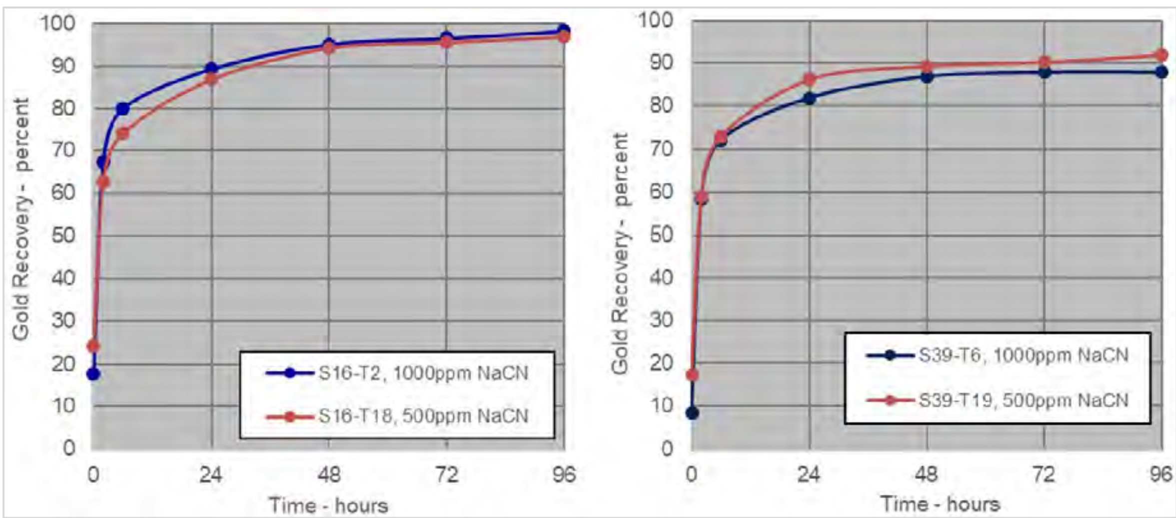
Most of the tests presented high extractions under the elevated pH conditions applied to the Sona Hill variability samples and blends thereof. Mid to high 80% and low 90% extractions being achieved.

For the various master composite and saprolite blends, the extractions observed were in line with the expected leach outcomes as a function of the blend ratio and the component extractions alone. Non-Sona Hill samples being leached at the more conventional pH of nominally 10.

A notable exception was a blend of the Gold Var bulk sample and the SAP-MC (test 32) that reported a lower than expected recovery of 78.7% when the anticipated value should have been in excess of 90%. A high leach residue assay of 0.21 g/t Au was reported. This final residue result could well be erroneous and the cause of this inconsistency. The sample was not re-assayed and so this inconsistency remains unexplained.

Figure 13-26 presents two graphs comparing the leach kinetics of Sona Hill samples. One for variability sample S16 (tests 2 and 18, low Te) and the other S39 (tests 6 and 19, high Te). The plots show that while sample S16 reported a final recovery of around 97% at 96 hours, the recovery at 24 hours was around 88 to 90%. Similarly, sample S39 present a recovery of around 90% at 96 hours but was only of the order of 83% at the 24 hour mark.

The leaching is observed to be substantially complete at 48 hours and several percent points of addition leach extraction are achieved over the 24 to 48 hour period. There would be a case to design for a 48 hour leach time for these samples if there was adequate tonnage to justify the incremental capital and operating costs. However, with a blend of the Sona Hill Deposit with the Toroparu Deposit feed stocks being proposed, the practicality of the facility having a 48 hour residence time for these other feed stocks will be a function of economics at the time of detailed design.



Source: BML, 2020

Figure 13-26: Comparative leach kinetics

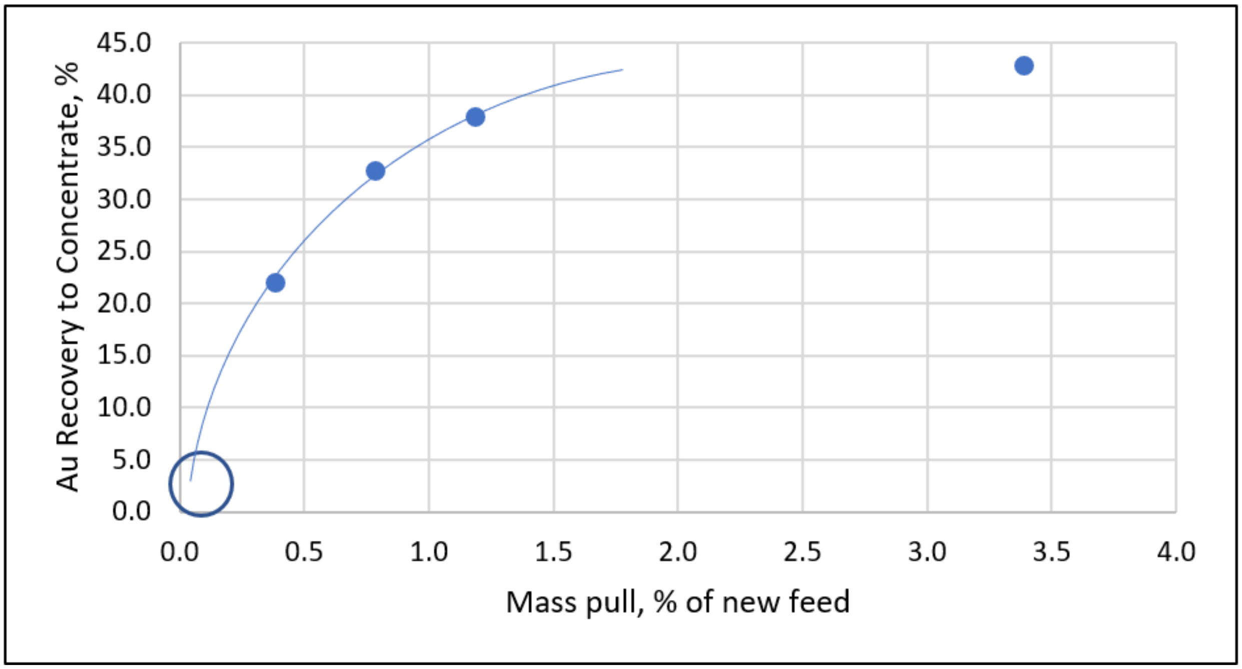
A gravity Mass-Recovery test was undertaken on the gold Bulk Composite where a Knelson concentrate was passed over a Mozley table to generate data relating mass pull to concentrate and the gold distribution. The results are summarized by Table 13-90.

Table 13-90: Gravity Mass-Recovery Test Work Summary

Product	Weight		Assay, g/t	% Distribution
	%	grams	Au	Au
Mozley Conc 1	0.4	7.8	101.1	22.0
Mozley Conc 1+2	0.8	16.9	69.5	32.7
Mozley Conc 1+2+3	1.2	23.4	58.0	37.8
Moz Con 1-3 + MozTail	3.4	67.8	22.7	42.8
Knelson Tail	96.6	1937.3	1.06	57.2
Recalc. Feed	100.0	2005.1	1.79	100

Source: BML, 2020

A graphical representation is provided per Figure 13-27. This plot shows how sensitive the gravity recovery is as the mass pull varies. While the test work presents mass pull values in the 0.5% to 3.5% range, the practical plant application typically provides mass pull values in nominally the 0.02% range as represented by the blue circle. The plot highlights that while +25% recovery values may be achieved in test work, the practical application may only result in values in the 5% to maybe 10% at best on feeds represented by this sample. Accordingly, design considerations in the plant carbon circuit need to be based on very low gravity recoveries at times such that the CIL will effectively see the mill feed grade and not a much lower gravity tail grade introduced to the leach as is suggested by the high mass pull test work.



Source: Data from BML, 2020

Figure 13-27 Recovery to gravity conc as a function of mass pull

Test work was conducted by leaching a number of sequential gravity concentrates. Results are summarized by Table 13-91. This work presented a relationship as to the gravity recovery as a function of mass pull to concentrate as well as presented a reducing leach extraction of concentrate as the mass pull increases.

As anticipated, this work highlights that as the mass pull is increased, the concentrate comprises more and more components that are not easily leached. Sulphide and potentially telluride associated gold being anticipated based on earlier mineralogical studies.

Table 13-91: Gravity Concentrate Sequential Leach Summary

Product	Weight		Assay – g/t	Dist'n – %	Extract'n – %
	%	grams	Au	Au	Au
Knelson Con 1	0.5	100.0	128	30.7	92.7
Knelson Con 2	0.5	100.8	38.3	9.3	79.6
Knelson Con 3	0.5	92.0	27.6	6.1	77.3
Knelson Tail*	98.5	19,707.2	1.14	53.9	53.9
Recalc. Feed	100.0	20,000.0	2.08	100	-

Measured Feed	2.50	

Source: BML, 2020

Carbon characterization tests (equilibrium and kinetics) were conducted on a leached sample of the gold Bulk Composite. This work was undertaken at high pH of 12.5 as previous carbon characterization work was performed at pH values of nominally pH 10. Elevated pH having deleterious impact on carbon characteristics.

Equilibrium values were measured for both a solution sample and then a slurry sample. The results are summarized by Table 13-92. It is noted the slurry equilibrium values are significantly lower than those achieved by solution alone. Design should apply the slurry values when the carbon circuit is assessed.

Table 13-92: Carbon Equilibrium Values

Test	Sample	Equilibrium Loading Au g/tonne Carbon	Solution – Au g/tonne
N1-N5	T36 Bulk Composite Solution	3,753	1.00
		2,565	0.50
		1,550	0.20
N6-N10	T36 Bulk Composite Slurry	2,876	1.00
		1,725	0.50
		877	0.20

Source: BML, 2020

Carbon kinetic values are summarized by

Table 13-93. These values are considered very low compared to the values achieved at more typical pH values of 10. This work suggests that the carbon circuit design will need to be flexible in that at times loadings can be expected to be in the typical range found in many CIL plants when processing the bulk of the Toroparu Deposit feed types. However, when processing Sona Hill material at elevated pH, the elution rate can be expected to be ramped-up considerably due to poor carbon kinetics. This work indicates that the elution method would be best serviced by an Anglo American Research Laboratories (AARL) type system given this type of facility as the best turn-up/turn-down ability.

Table 13-93: Carbon Kinetic Values

Sample	Feed Solution Au (ppm)	Flemming Kinetic Constants	
		k (hr^{1})	n
T36 Bulk Composite ground to 58um K80	1.45	41.7	0.79

Source: BML, 2020

Continuous cyanide detoxification test work was undertaken on a leached sample of the gold Bulk Composite. The work showed the SO_2/air process is amenable to treating the slurry but did require copper dosing at 30 g/L of slurry to catalyze the reaction. A residence time of 90 minutes was applied and a dose rate of 5.5 g SO_2/g CN_{WAD} was applied. There is scope to reduce the SO_2 dose given the dose applied is on the upper end of typical commercial application rates, but this would require additional test work to confirm.

For Sona Hill blends, the high pH discharging the CIL has the potential to impact on the cyanide detoxification. As effective detoxification was observed to occur at a lower pH, it has not been established if the dose of SO_2 will be adequate to reduce the pH to this level.

Static and dynamic settling tests were performed on the Bulk composite and static tests on saprolite blends. Flocculant testing suggested Magnafloc MF10 was preferable to the previously selected MF1011 for the fresh ore-saprolitic blends at the high pH of 12.5. However, MF1011 was suited to the other settling duties where saprolite was not present.

Saprolite blends did not provide for high pulp densities on final static settling. Values of the order of 45 to 52% solids being achieved. This is a consideration for tailings storage design parameters. It should be remembered the saprolite portion of these blends is higher than the current blend proposed by the PEA production schedule.

Fresh ore blends gave improved results, but the values were still lower than may have been expected. However, the tests were conducted at a neutral pH and the results are not relevant to design as a consequence.

The gold Bulk Composite was the only sample subjected to dynamic tests and this was run at a pH of 11. High flux rates were achieved commensurate with previous settling test work. The pulp densities achieved were moderate with a maximum of 58.8% solids achieved with a high flocculant dose of 55 g/t with good overflow clarity. These values are in alignment with previous testing and flocculant demands previously suggested as a basis for design per the earlier FLS test work of 2014.

Yield stress determinations for the Bulk composite did not present any high stresses at elevated pulp densities. These values typically applied by thickener vendors for rake design.

Sheared viscosity testing was conducted on a number of saprolite based blends as well as saprolite alone. Various shear rates and pulp densities evaluated to assess the impact on different unit processes in a process plant. Results are summarized by

Table 13-94.

Table 13-94: Viscosity Testing Results

Sample ID	Density % Solids	Shear Rate, s-1							
		100 to 51	28 to 33	15 to 18	9 to 11	5 to 6	3 to 4	2	1
		Viscosity – cps							
GRDT-MC/SAP Blend	40	115	70	48	63	93	143	222	395
	50	122	111	174	270	435	714	1166	2047
	60	690	980	1513	2548	4191	6726	12100	19620
GRDT-QZ/SAP Blend	40	-	199	91	61	56	79	105	157
	50	122	80	103	153	241	373	624	1061
	60	294	424	706	1105	1790	2989	4829	5923
SAP Composite	40	113	152	260	412	698	1166	2026	3552
	50	669	979	1712	2739	4614	7452	11030	18780

Source: BML, 2020

The viscosity work suggests:

- For shear rates associated with pumping, there should be no viscosity issues. System design will need to consider elevated viscosity on thickener underflows.
- Saprolitic blends present elevated viscosity at low shear rates. These will need to be considered for thickener rake performance, low shear piping duties and notably, for inter-tank screen, trash and carbon safety screen and launder duties.
- Milling and classification should be conducted at lower pH and then the CIL plant feed pH should be adjusted post these unit processes when processing Sona Hill feeds. This implies the CIL plant pH control system will require significant milk of lime addition at the leach tanks.

These points noted, there are mitigating considerations:

- Flow considerations post cyanide detox will not be as onerous as the pH will have been reduced to nominally 9 whereas the test work was conducted at pH 11. Lower viscosities can be expected at full-scale plant conditions.
- Similarly, the pH range will normally be of the order of 10 unless Sona Hill feeds are being processed. Lower viscosities can again be expected.
- The blends tested comprise 38% saprolite. The proposed blends are 30% saprolite in year 1 and 15% thereafter. Consequently, the influence of saprolite on viscosity will generally be less that determined above.

13.13 Conclusions and Recommendations

There has been a substantial amount of metallurgical test work undertaken on a spread of samples originating from the deposits over the last decade.

The test work has confirmed that flotation and free milling cyanidation process routes are capable of achieving high recoveries of both gold and copper as well as payable silver values.

The processing strategies have influenced the test work programs over the project development period. These blends will have their own characteristics, but test work results are generally observed to align with the expected values for the blends.

Not all blends from all sources have been tested. This is in part due to sample availability but also due to the ongoing development of the project over the intervening period. Designs will need to consider how this may be managed in the full-scale processing plant.

Saprolitic material adds to viscosity considerations as well as mill operating practices. Being soft, the power demand in the milling circuit is reduced and so high saprolite blends may allow elevated processing rates of hard ores. However, classification operations will need to be monitored if viscosity gets too high.

The Sona Hill fresh blends have differing characteristics for leaching of gold values. Namely, the presence of auriferous tellurides that are slower leaching, require a finer grind and elevated pH to achieve high extractions. Blending of Sona Hill fresh material with other material types adds to the operating cost of the other ores (elevated quicklime demands). At the same time blending results in lower extractions from the Sona Hill component than would be achieved if the Sona Hill were processed alone at lower throughput so as to achieve a finer grind.

There is an upside scenario that has not been tested for the Sona Mill material, and that is elevated temperature leaching. In general, conventional grinding and processing via CIL will be undertaken at temperatures of +30 degrees Celsius and as high as 40 degrees in warmer climates with a high recirculation of process water from tails thickening. The benefit of temperature on telluride leaching is usually quite marked, and so leach rates can be expected to be elevated.

The high pH regimes needed to process Sona Hill material will influence the carbon circuit performance, and this needs to be considered in design and process selection of the elution technology. It is also a consideration for the cyanide detoxification process, and the design and capacity of reagent systems. Notably pH modifier.

Early work conducted by SGS reported the presence of tellurides in the samples tested at that time. These samples originated within the central portion of the Toroparu Deposit. Slow leaching was observed for some of these samples, which aligns with the presence of auriferous tellurides. However, the test work conducted post these SGS programs (2014 and later) have not presented slow leaching scenarios and have supported a reduced leach residence time, which is the current basis of the PEA CIL facility. This inconsistency has been highlighted by the more recent Sona Hill test work and warrants further evaluation.

Cyanide soluble copper has been identified as something that needs to be managed. As the project develops and particularly during operation, the anticipated blend ratios may well change. Consequently, the flowsheet will need some flexibility to address this. The impact of copper on the carbon circuit has not been evaluated at a range of soluble copper concentrations. There is the opportunity with new sample availability to conduct some work to explore this if deemed justified.

The introduction of single stage SAG milling (SSAG) as a comminution option has been assumed to be practical. However, there is limited SAG design data available, and that which is available, shows that the material tested is very competent. Additional SAG parameter determinations are needed to give confidence in SAG selection, grind achievement and suitability for the various blends proposed.

The following recommendations are made:

- Elevated temperature leach tests on Sona Hill variability (high Te) samples.
- Review of the test work data and establish if tellurides in earlier work (pre-2014) may have been responsible for slow leaching samples and if this observation may influence CIL design.
- Expanded variability testing to better understand cyanide soluble copper characterization, leach and flotation behaviour based on revised feed blends and sources.
- Perform carbon characterization tests under variable cyanide soluble copper concentrations.
- SAG mill parameter testing and in the interim, circuit modelling based on known parameters.

A test work program had commenced at the time of writing. This program is using a selection of new samples freshly drilled and will address a number of the points listed above. The intention is to also recover Sona Hill samples from storage and address temperature opportunities.

With regard to progressing into detailed design of the process plant, it will be necessary for the encumbered engineer to review all of the detailed test work programs and reports to confirm and/or establish the relevant design criteria for the project as defined at the time. Namely, the ore blends and ratios that are proposed with due consideration as to how recoveries will be impacted and what design contingencies will be required.

Additionally, there will be trade-off scenarios regarding location of thickeners in the circuit, justification of leach duration, gravity effort and tank farm and flotation cell configuration optimization as a function of the blends and throughput proposed. Some of these trade-offs may require additional test work as the design progresses.

14 MINERAL RESOURCE ESTIMATE

14.1 Drill Hole Database

The Mineral Resource Estimate was calculated from two main databases for the Project, one for the Toroparu Deposit and another for the Sona Hill Deposit. Both complete databases are comprised of a total of 709 diamond drill holes and three trenches consisting of 199,996 m. This includes:

- Toroparu Deposit has 528 diamond drill holes consisting of 178,491 m and three trenches comprised of 655.3 m completed between 2006 and 2021, and
- Sona Hill Deposit area has 181 diamond drill holes consisting of 20,850.0 m completed between 2012 and 2018.

The Mineral Resource Estimate was prepared by Nordmin following a two-phase diamond drill program in 2020-2021 which comprised a total of 20,750 m in 114 drill holes. The new drill hole assays were reviewed and fully validated by Nordmin.

A drilling summary can be seen in Table 14-3, and images of the drill holes used in the 2021 Mineral Resource Estimate can be seen in Figure 14-1 and Figure 14-2.

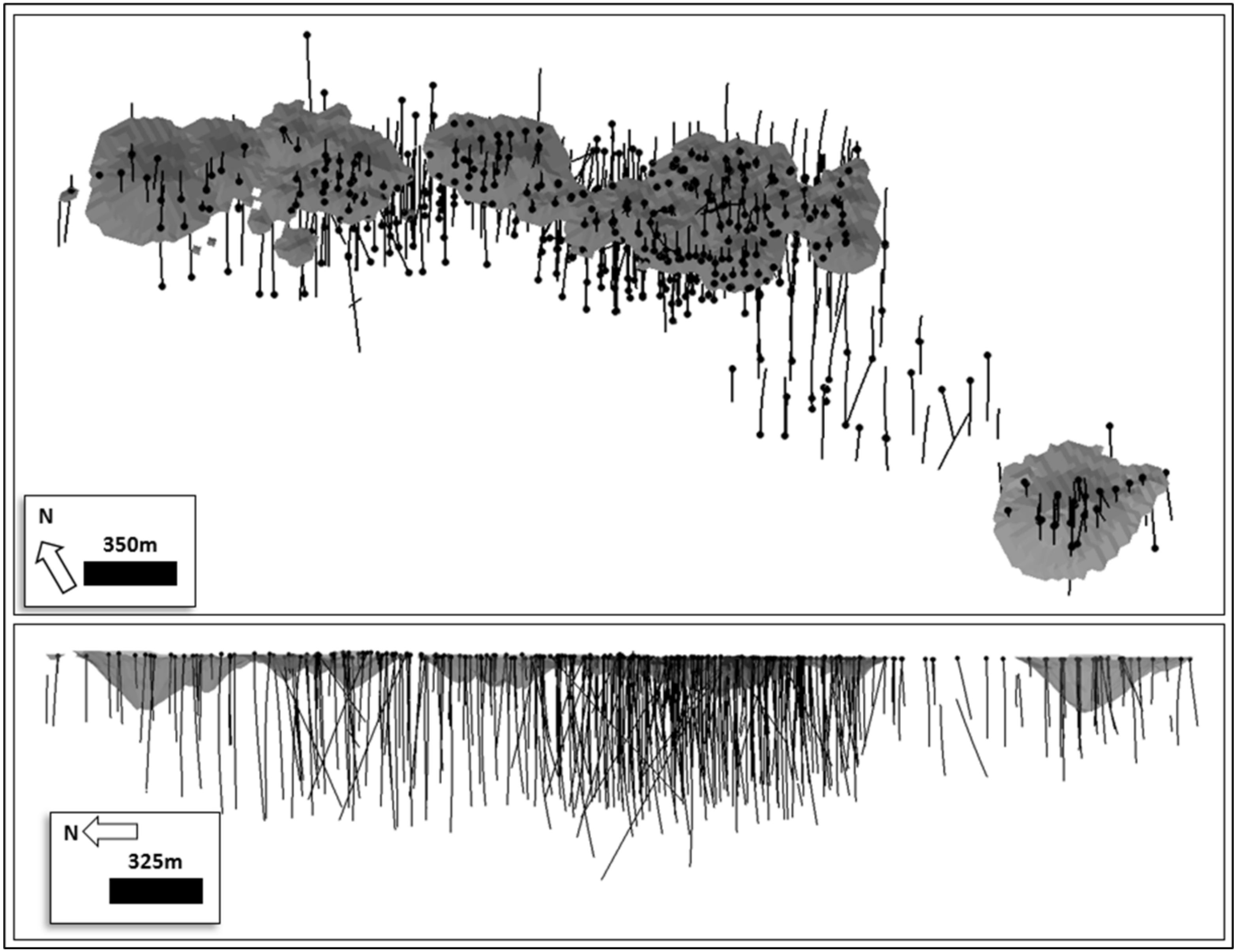


Figure 14-1: Drill holes, Toroparu Deposit plan, and long section view

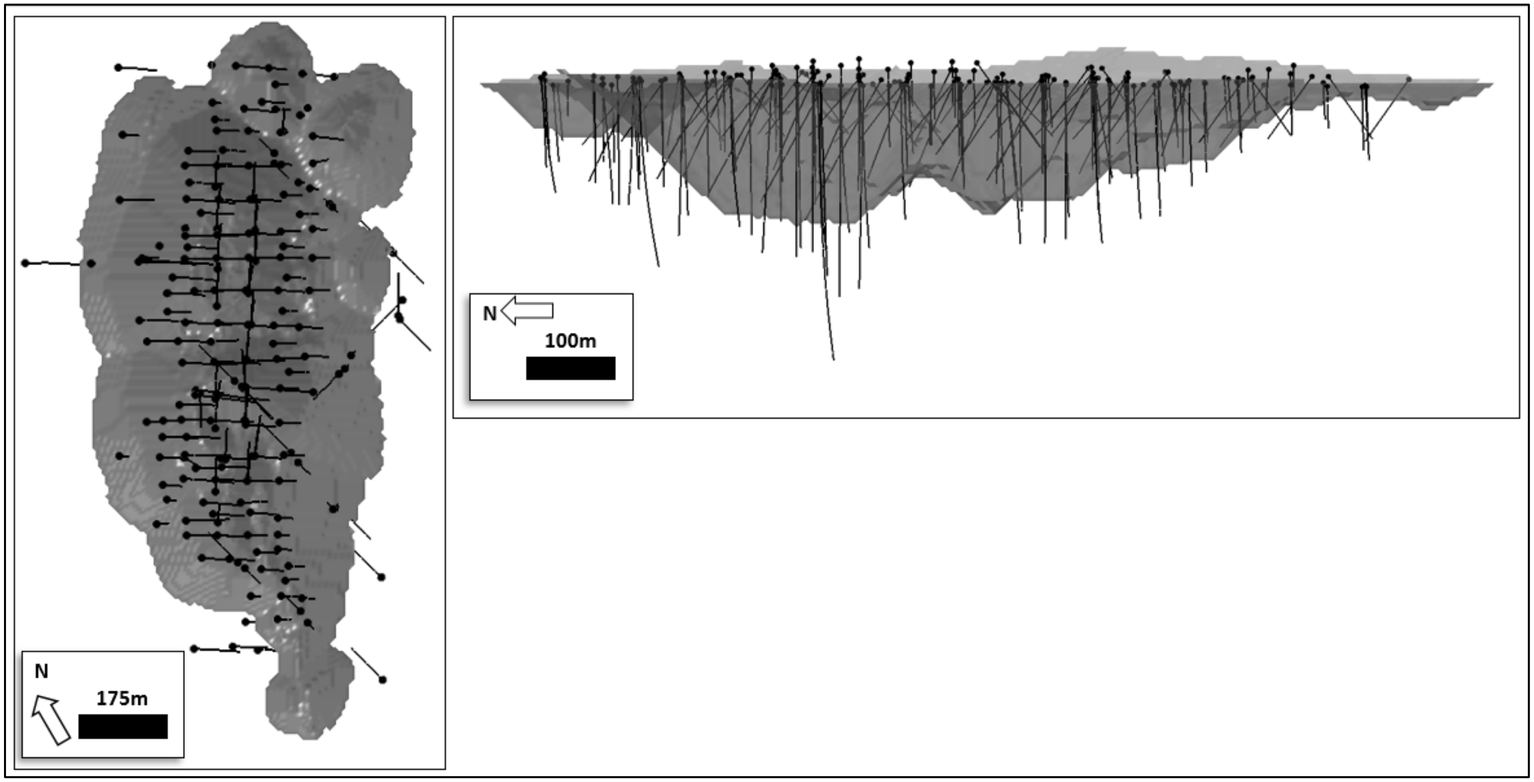


Figure 14-2: Drill holes, Sona Hill Deposit plan, and long section views

Table 14-1: Project Drilling Summary

Year	Deposit	Series	Hole Type	Count	Length (m)	Assay Count
2006	Toroparu	TPD001 to 002	DDH	2	518.0	517
2007	Toroparu	TPD003 to 022	DDH	21	6,749.0	4,422
2008	Toroparu	TPD023 to 029	DDH	7	2,990.5	1,979
2009	Toroparu	TPD030 to 050	DDH	22	10,078.2	6,621
2010	Toroparu	TPD050 to 128	DDH	75	43,515.1	30,087
2011	Toroparu	GT01 to 05	DDH	5	2,302.5	1,519
	Toroparu	TPD129 to 326	DDH	208	80,236.3	53,488
2012	Toroparu	TPD327 to 468	DDH	151	34,385.4	20,593
	Toroparu	TPR001 to 081	DDH	81	6,329.9	3,493
	Toroparu	TPT001 to 003	Trench	3	655.3	143
	Sona Hill	SOD001 to 005	DDH	5	811.0	549
	Sona Hill	SOR001-030	RC	30	2,968.9	944
2014	Toroparu	BH-165	AC	1	30.0	10
	Toroparu	DH-MF-01 to 17	AC	17	758.0	255
	Toroparu	DH-TS-01 to 11	AC	11	467.6	161
	Toroparu	Q2	AC	1	20.0	7
	Toroparu	TPHY01 to 02	AC	2	54.0	18
	Toroparu	TPC001-171	RC	172	6,161.5	2,077
2015	Sona Hill	SOD006 to 042	DDH	37	3,780.5	2,711
2016	Sona Hill	SOD043 to 109	DDH	67	7,994.2	5,539
2017	Sona Hill	SOD110 to 131	DDH	22	3,099.7	1,997
2018	Sona Hill	SOD132 to 184	DDH	53	6,277.7	4,264
2020	Toroparu	TPD469 to 539	DDH	72	10,256.3	8,254
2021†	Toroparu	TPD540 to 555†	DDH	15†	2,649.4†	2,485†
Total Toroparu				**864**	**207,639**	**135,612**
Total Sona Hill				**214**	**24,932**	**16,004**
TOTAL				**1,078**	**232,571**	**151,616**

† As of the cut-off date of March 18, 2021

Table 14-2 summarizes the drill holes and samples in the Toroparu and Sona Hill Deposit drilling databases.

Table 14-2: Total Toroparu and Sona Hill Deposit Drilling Database Summary

	Toroparu Deposit		Sona Hill Deposit
	Diamond Drill	**Trenches**	**Diamond Drill Holes**
Number of Drill Holes and Trenches	893	3	214
Number of Survey Records	5,619	34	814
Number of Lithology Records	11,547	0	5,146
Number of Gold Assay Records	137,268	143	16,004
Number of Copper Assay Records	27,610	143	16,010
Number of Silver Assay Records	27,789	0	2,711
Number of ICP Assay Records	27,753	0	2,711

14.2 Geological Domaining

Nordmin, through an interactive process with the Company, undertook a full re-examination of the mineralogical, lithological, structural, and geochemical correlations influencing gold mineralization within the Project. The review concluded that:

- The previous modelling of the mineralization utilizing a single implicit lower grade 0.2 g/t gold shell did not identify nor isolate the structurally controlled higher-grade domains that exist throughout the project area.
- The previous interpretation was not representative of the deposit type nor the geological controls of mineralization that support both lower grade and higher-grade mineralized domains.
- Each domain and corresponding sub domains required extensive modelling of the higher-grade structural domains, which control the higher-grade mineralization within the encapsulating lower grade mineralized domain.

Therefore, the previous Mineral Resource Estimate did not include the well defined structural geological architecture required to support the geometry of multi-generational lower and higher-grade mineralization. As a result, the previous Mineral Resource Estimate created a large, lower grade open pit resource that was ranked as one of the largest bulk tonnage resources in the Americas. Nordmin's re-examination of the project data delineated the geological architecture and structural controls that commonly control the placement of higher-grade mineralization within the majority of the Archean aged gold deposits around the world, for example, the Red Lake, Timmins, Kirkland Lake, and Côté Lake mining camps in Ontario, Canada. The 2020 and 2021 20,750 m (114 hole) drill program further verified the location and structural relationship between the lower and higher-grade mineralization domains located within the previously defined disseminated lower grade mineralized halo along the 4 km Toroparu trend and for the Sona Hill Deposit.

The Project consists of two deposits referred to as the Toroparu Deposit and the Sona Hill Deposit. The Sona Hill Deposit is located approximately 3.2 km southeast of the Toroparu Deposit (Figure 14-3).

Nordmin incorporated the various geological, structural controls to support the various gold, copper, and silver mineralization styles, and their associated geochemistry. The block model utilized explicit modelling of mineralized structures present in the deposit areas to support the Mineral Resource Estimate. These models incorporate the geologic and structural controls of gold mineralization, the style of mineralization, and its associated geochemistry. Nordmin's opinion is that utilizing the explicit modelling approach minimizes risks compared to using implicit modelling for the Project. The Toroparu Deposit consists of multiple geographical areas, including the Main, NW, and SE Areas. Each of these areas was separated into three domains, including a Background Low-Grade Domain (Low-Grade), a northwest-southeast structural domain (NW-SE), and an east-west structural domain (E-W) which crosscuts the NW-SE domain (Figure 14-4 and Figure 14-5). The Toroparu Deposit Model utilized significantly more drill holes and several assays than the previous 2018 Mineral Resource update. Within the Sona Hill Deposit, there are three similar domains consisting of a Background Low-Grade Domain (Low-Grade), a northwest-southeast domain (NW-SE), and an east-west domain (E-W) which crosscuts the NW-SE domain (Figure 14-6).

Figure 14-4, Figure 14-5, and Figure 14-6 depict the orientations and relationships between the Low-Grade and Higher-Grade Domains. The intersection of the NW-SE and E-W structures creates zones of wider and higher-grade gold mineralization than in the structures themselves. These structural intersections occur over a consistent and repeatable pattern that enriches gold, silver, and copper mineralization throughout the deposits. The recognition of these patterns supports the combination of open pit and underground mining methods that form the basis of the Mineral Resource Estimate. Figure 14-7 is a cross-section showing mineralization in the Main Area of the Toroparu Deposit.

A summary of all Domains can be seen in Table 14-3.

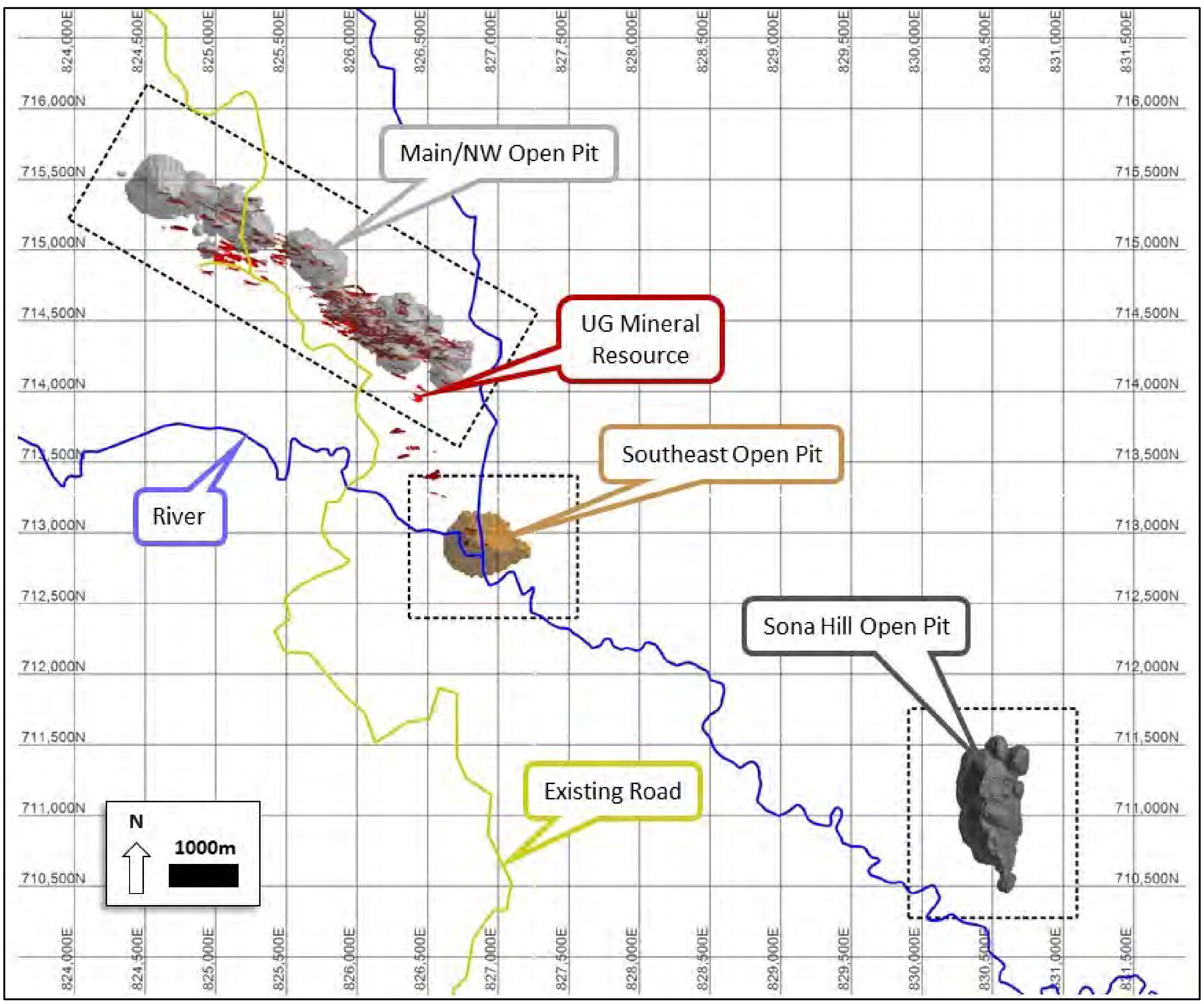


Figure 14-3: Project plan overview

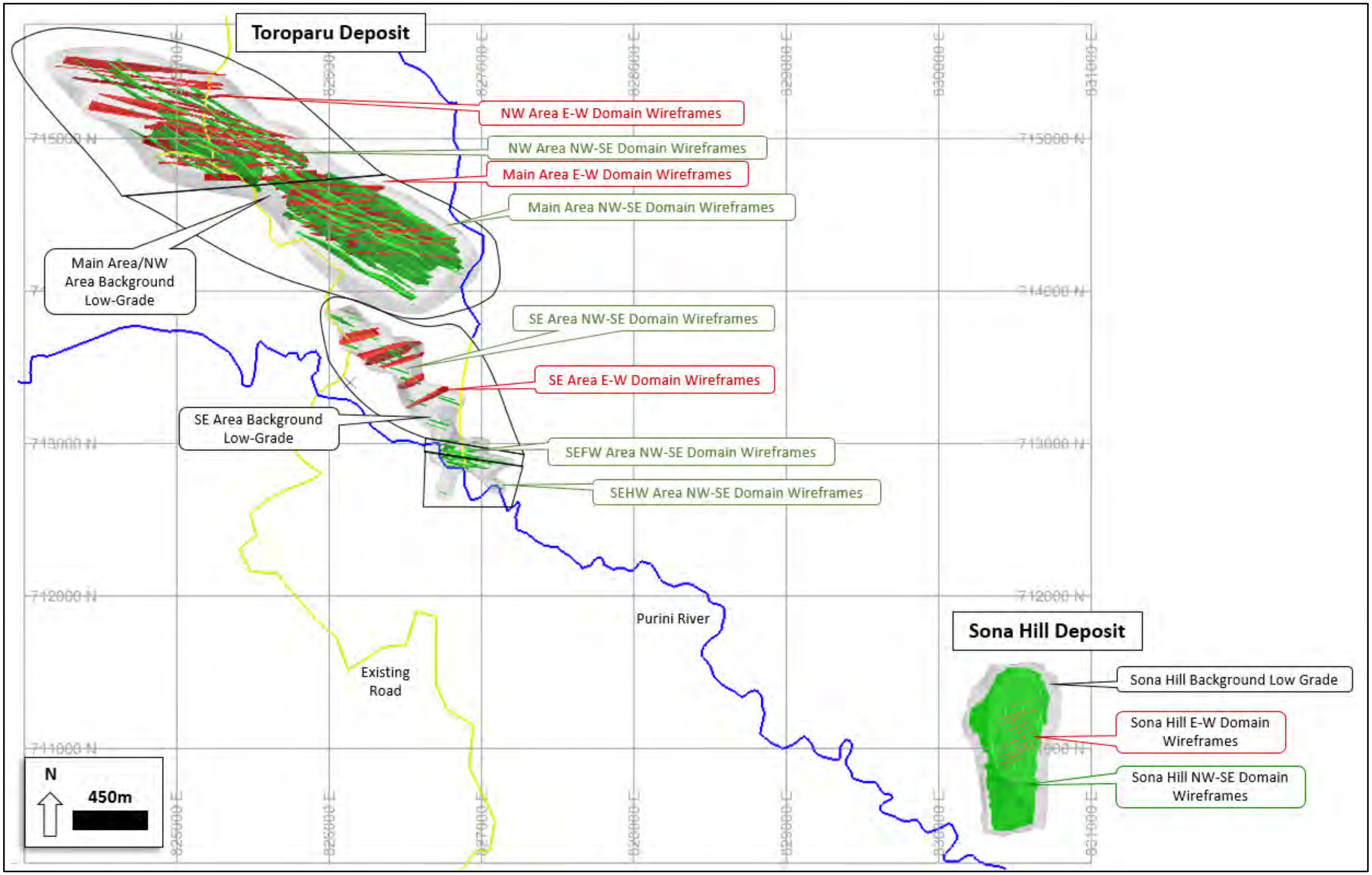


Figure 14-4: Toroparu and Sona Hill Deposits, plan overview

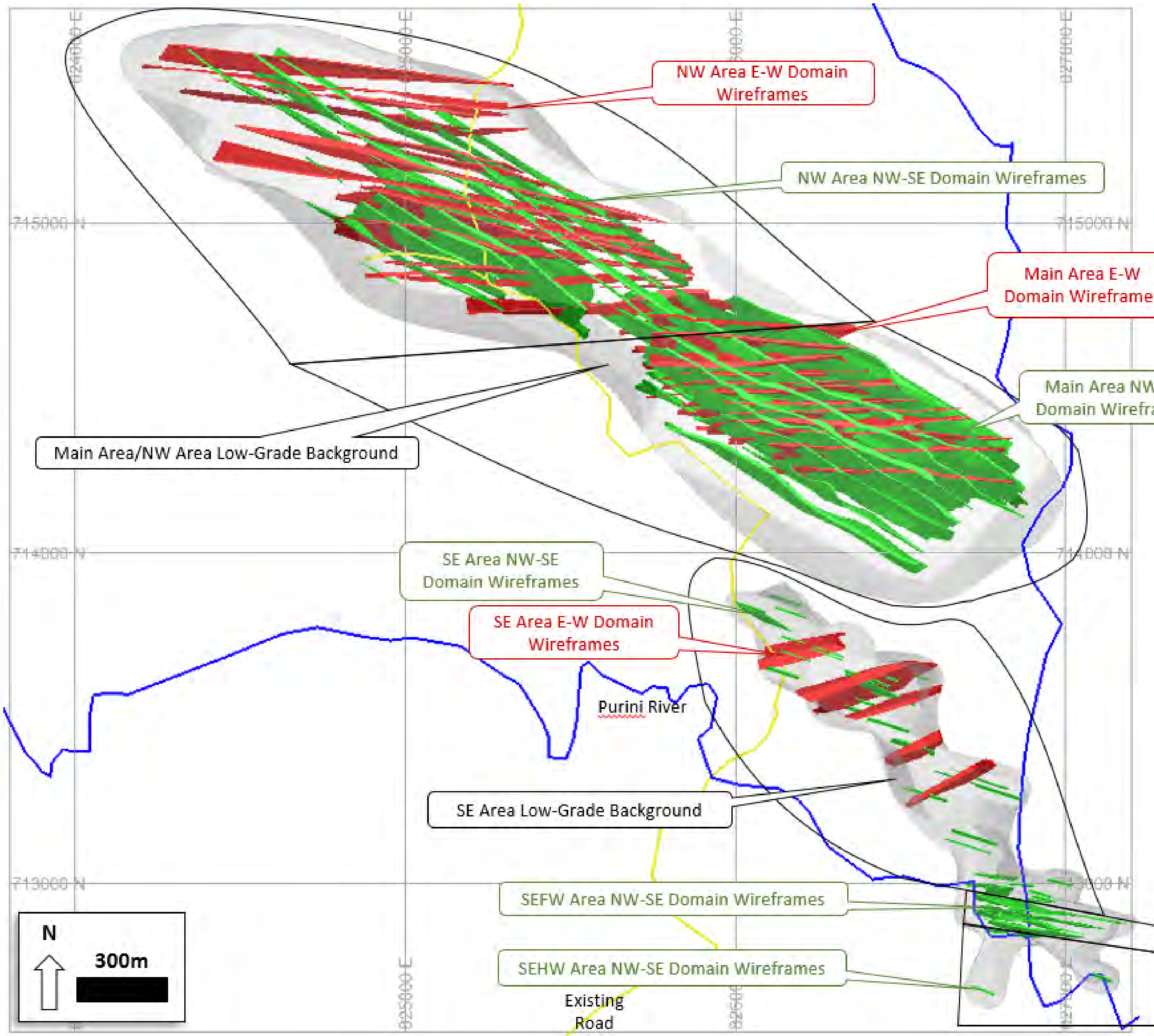


Figure 14-5: Toroparu Deposit overview with Areas

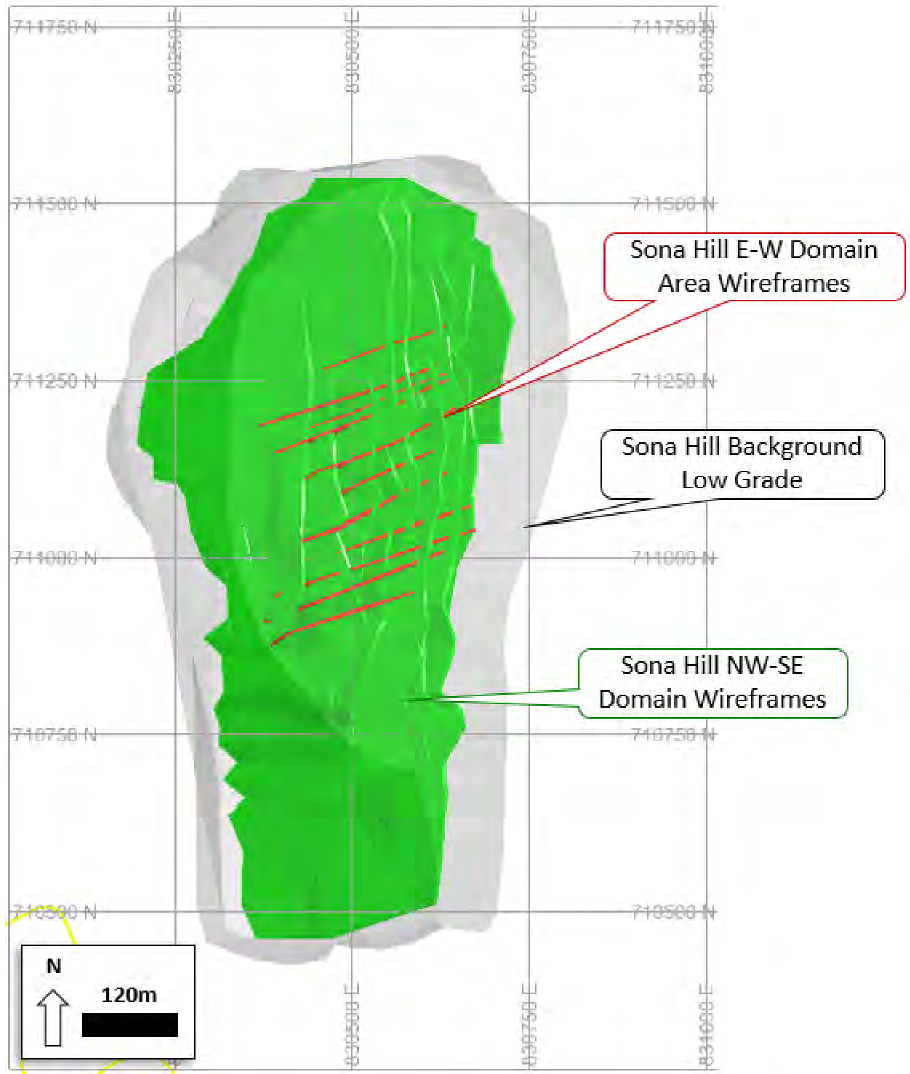


Figure 14-6: Sona Hill Deposit overview

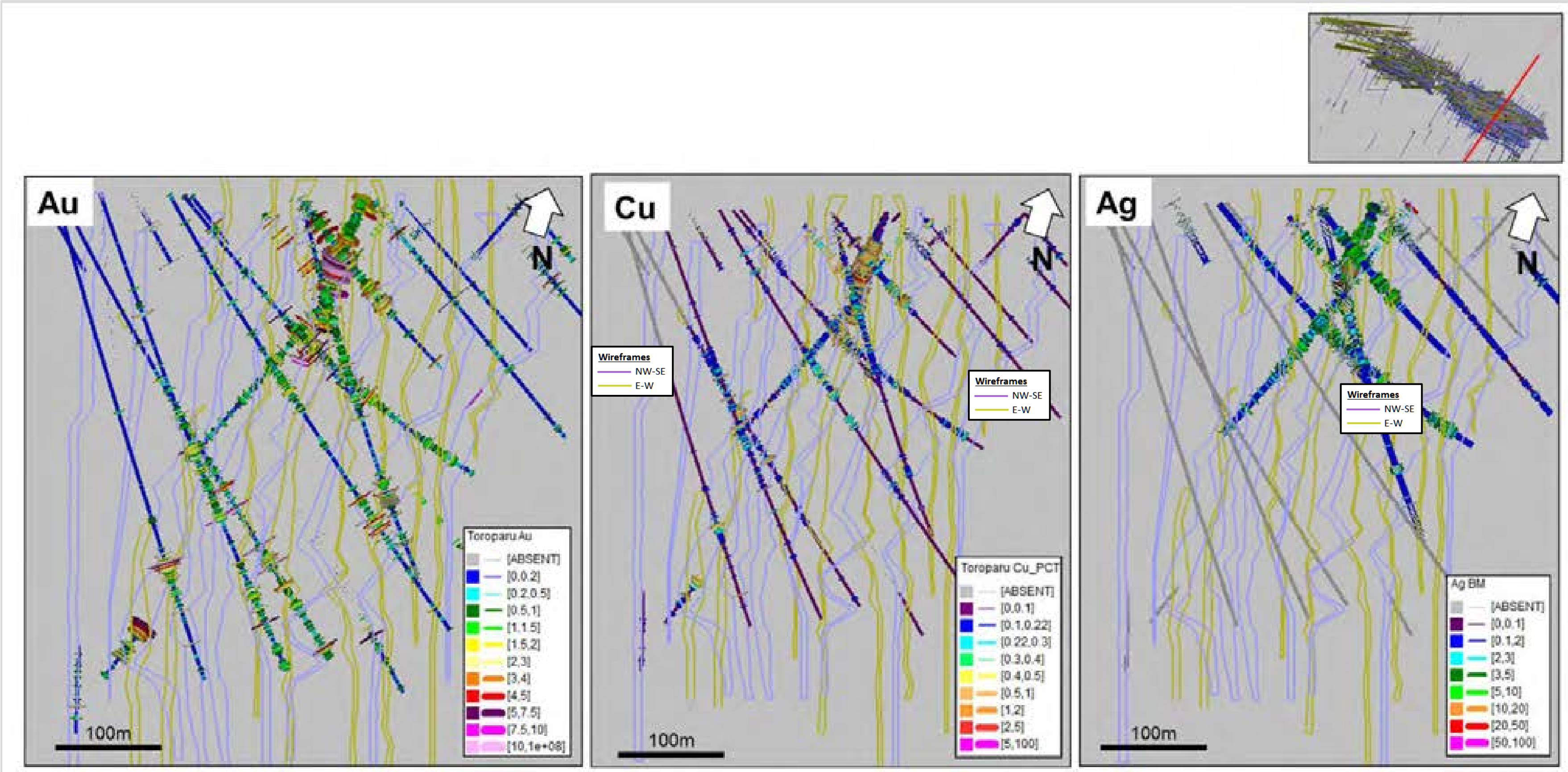


Figure 14-7: Cross-section of mineralization within the Toroparu Deposit, Main Area

Table 14-3: Domaining

Deposit	Area	Domain	Wireframe Count	Wireframes
Toroparu	Main/NW Background		1	1
	NW	NW-SE	11	1-10,12
		E-W	16	1-16
	Main	NW-SE	9	1-9
		E-W	12	1-12
	SE/SEHW/SEFW Background		1	1
	SE	NW-SE	24	2-11,13-26
		E-W	5	1-5
	SEHW	NW-SE	9	2-10
	SEFW	NW-SE	8	2-4,6-10
Sona Hill		Background	1	1
		NW-SE	14	1 through 14
		E-W	11	1 through 11

For the Toroparu and Sona Hill Deposits, the mineralized wireframes (NW-SE and E-W) were modelled using the following criteria:

- An approximate cut-off grade of 1.0 g/t gold.
- Structural model: Structural trends were observed while developing the model.
    - The Company identified two major quartz vein orientations in general logging and oriented core analysis:
        - QZO: These are typically greater than 30° to core axis, and
        - QZP: These are typically less than 30° to core axis.
    - A structural stereonet analysis was performed on each of these quartz vein datasets, which determined that:
        - The QZO veins correlate directly to the E-W structural Domain wireframes, and
        - The QZP veins correlate directly to the NW-SE structural Domain wireframes.
    - See Section 7.3.1 (Toroparu Deposit) and 7.3.2 (Sona Hill Deposit) for more information.
- Wireframes were permitted to follow geological and lithological boundaries and trends where appropriate.

The Background Low-Grade Domain wireframes were modelled using the following criteria:

- An approximate cut-off grade of 0.1 g/t gold.
- Structural model: Structural trends were observed while developing the model. The Background Low-Grade wireframes were developed to encompass the E-W and NW-SE Domain wireframes fully. See Section 7.3.1 (Toroparu Deposit) and 7.3.2 (Sona Hill Deposit) for more information.
- Wireframes were permitted to follow geological and lithological boundaries and trends where appropriate.

Wireframes were primarily created on 15 m to 25 m vertical sections depending on drill density. They were adjusted on plan views to edit and smooth each wireframe where required. When not cut-off by drilling, the wireframes terminate at plunge and depth.

No wireframe overlapping exists within Domains, and all are independent of each other. Each Domain was developed mutually exclusive to other Domains. East-West (E-W) and Northwest-Southeast (NW-SE) wireframes overlap, and both Domains are encompassed by Background Low-Grade Domains. These wireframes overlap from a modelling perspective but not from a resource, grade, and tonnage perspective. For every Domain Area, each domain wireframe was independently estimated, and the resulting block models were consolidated ("added") in a determined order as follows to create the final block models:

- NW-SE Domain wireframes are added over the Background Low-Grade Domains.
- E-W Domain wireframes are added over the block model resulting from the step above.

14.3 Exploratory Data Analysis

The exploratory data analysis was conducted on raw drill hole data to determine the nature of the gold distribution, correlation of grades within individual rock units, and the identification of high-grade outlier samples. Nordmin used a geostatistical package (X10 Geo) to complete various descriptive statistics, histograms, probability plots, and XY scatter plots to analyze the grade population data. The findings of the exploratory data analysis were used to help define modelling procedures and parameters used in the Mineral Resource Estimate.

Descriptive statistics were used to analyze the grade distribution of each sample population, determine the presence of outliers, and identify correlations between grade and rock types for each Background Low-Grade Domain, NW-SE Domain, and E-W Domain.

Individual drill hole tables (collar, survey, assay, etc.) were merged to create one single master desurveyed drill hole file. The process splits assay intervals to allow for all records in all tables to be included. Assay wireframe counts are found in Table 14-4, Table 14-5, and

Table 14-6 and are based on analysis of this master file; counts will differ when compared with the original data.

Table 14-4: Assays by Wireframe for the Toroparu Deposit, Main, and NW Areas

Deposit	Area	Domain	Wireframe	Au Sample Count	Cu Sample Count	Ag Sample Count
Toroparu	Main/NW Background		1	81,521	79,899	18,449
	NW	NW-SE	1	12	12	0
			2	61	61	0
			3	106	103	0
			4	195	195	4
			5	253	253	3
			6	343	343	9
			7	628	628	48
			8	454	448	65
			9	471	467	105
			10	311	308	86
			12	1	1	0

Deposit	Area	Domain	Wireframe	Au Sample Count	Cu Sample Count	Ag Sample Count
		E-W	1	266	266	0
			2	47	47	0
			3	227	221	8
			4	22	22	0
			5	76	76	0
			6	493	489	16
			7	540	536	76
			8	288	288	62
			9	511	511	43
			10	356	356	34
			11	122	122	13
			12	74	74	15
			13	164	163	14
			14	205	205	34
			15	110	110	0
			16	64	64	3
	Main	NW-SE	1	628	627	93
			2	162	154	35
			3	1,169	1,164	397
			4	1,014	1,014	503
			5	2039	2038	1,021
			6	3,497	3,494	1,856
			7	1,889	1,876	654
			8	1,600	1,580	352
			9	370	359	49
		E-W	1	542	542	179
			2	497	487	165
			3	614	609	251
			4	898	898	420
			5	1,824	1,824	1,273
			6	452	452	166
			7	937	937	535
			8	516	516	285
			9	317	313	121
			10	607	604	212
			11	504	504	110
			12	425	425	60

Table 14-5: Assays by Wireframe for the Toroparu Deposit, SE, SEHW, and SEFW Areas

Deposit	Area	Domain	Wireframe	Au Sample Count	Cu Sample Count	Ag Sample Count
Toroparu	SE/SEHW/SEFW Background		1	5,217	5,204	0
	SE	NW-SE	2	5	5	0
			3	6	6	1
			4	3	3	0
			5	2	2	0
			6	4	4	0
			7	1	1	0
			8	3	3	0
			9	4	4	0
			10	3	3	3
			11	4	4	4
			13	7	7	0
			14	3	3	0
			15	4	4	0
			16	4	4	0
			17	2	2	0
			19	2	2	0
			20	4	4	0
			21	2	2	0
			22	2	2	0
			23	3	3	0
			24	5	5	0
			25	3	3	0
			26	5	5	0
		E-W	1	91	91	0
			2	224	224	19
			3	35	35	32
			4	125	125	0
			5	113	113	47
	SEHW	NW-SE	2	73	72	0
			3	6	6	0
			4	126	126	0
			5	15	15	0
			6	5	5	0
			7	6	6	0
			8	1	0	0
			9	3	3	0
			10	4	4	0
	SEFW	NW-SE	2	6	6	0
			3	32	32	0

Deposit	Area	Domain	Wireframe	Au Sample Count	Cu Sample Count	Ag Sample Count
			4	4	4	0
			6	237	237	0
			7	333	333	0
			8	4	4	0
			9	8	8	0
			10	2	2	0
			11	3	3	0
			12	2	2	0
			13	4	4	0

Table 14-6: Assays by Wireframe for Sona Hill Deposit

Deposit	Domain	Wireframe	Au Sample Count	Cu Sample Count	Ag Sample Count
Sona Hill	Sona Hill Background		12,314	12,314	2,266
	NW-SE	1	308	308	53
		2	480	480	106
		3	619	619	94
		4	212	212	43
		5	203	203	37
		6	11	11	0
		7	14	14	0
		8	67	67	7
		9	15	15	1
		10	152	152	19
		11	108	108	30
		12	21	21	4
		13	54	54	7
		14	47	47	8

Figure 14-8 through Figure 14-12 outlines the histogram and probability plots for the Toroparu Deposit NW and Main Areas. Figure 14-13 through Figure 14-15 outlines the plots for the Toroparu Deposit SE Area. Figure 14-16 through Figure 14-18 outlines the plots for the Sona Hill Deposit.

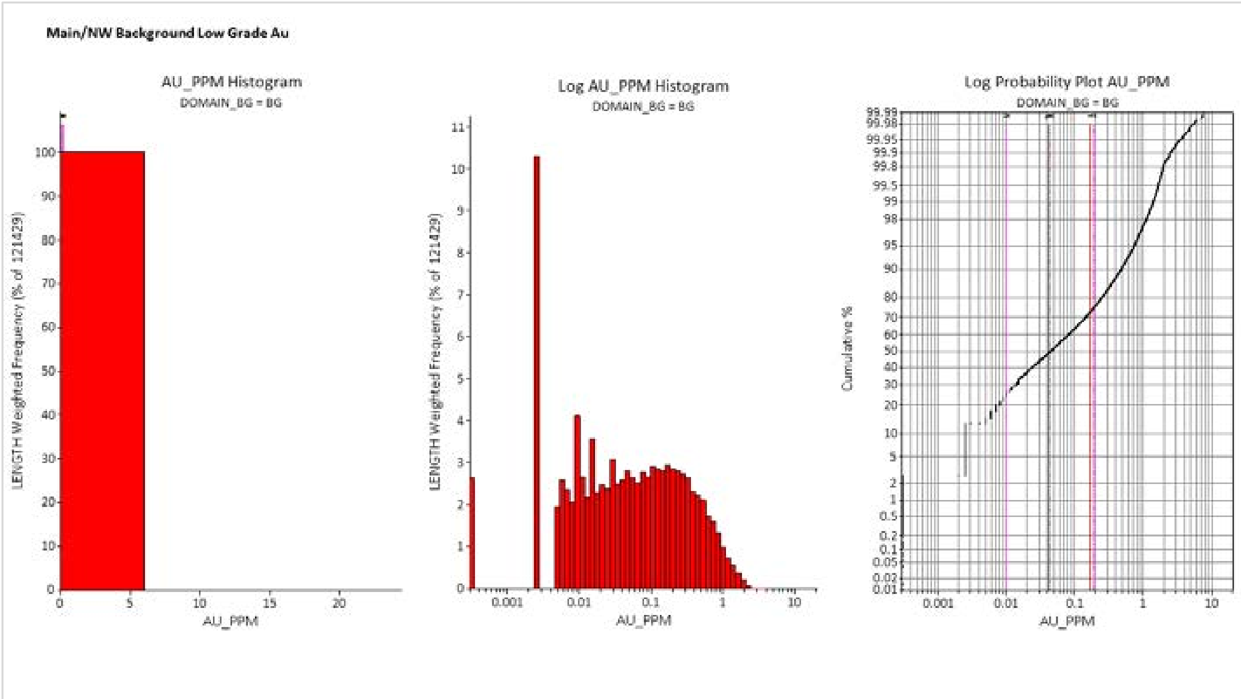


Figure 14-8: Gold histogram and probability plots for the Toroparu Deposit Main/NW Area Background Low-Grade Domain

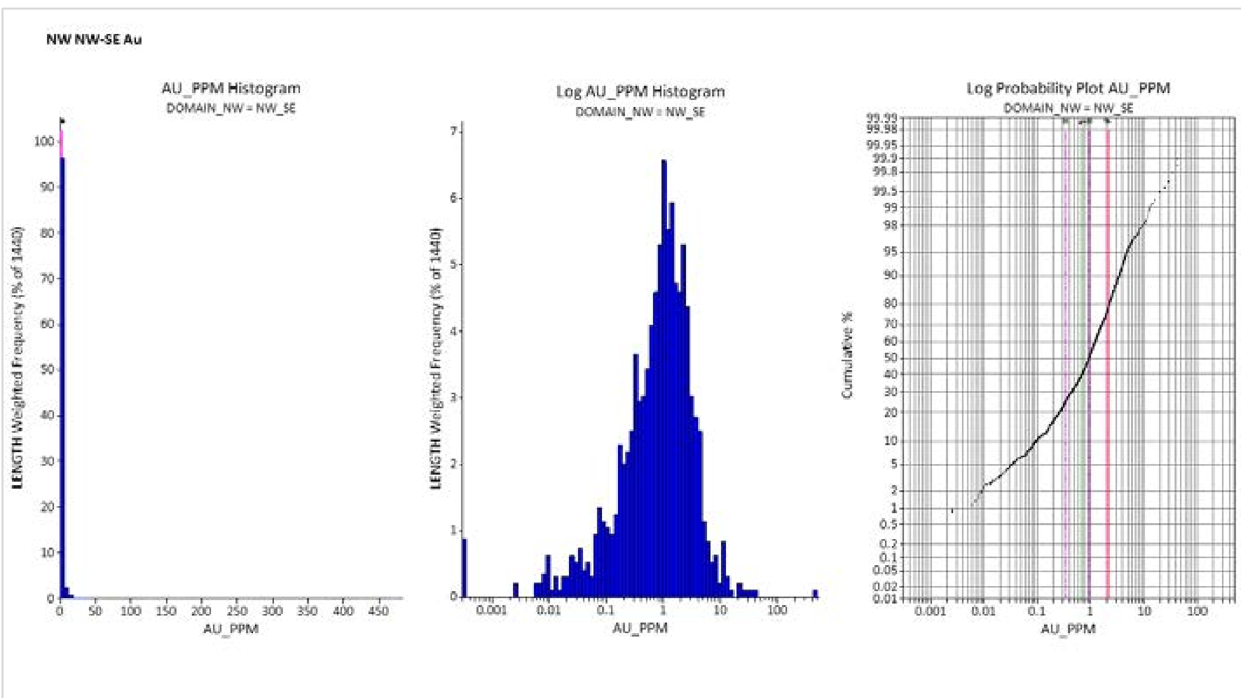


Figure 14-9: Gold histogram and probability plots for the Toroparu Deposit NW Area, NW-SE Domain

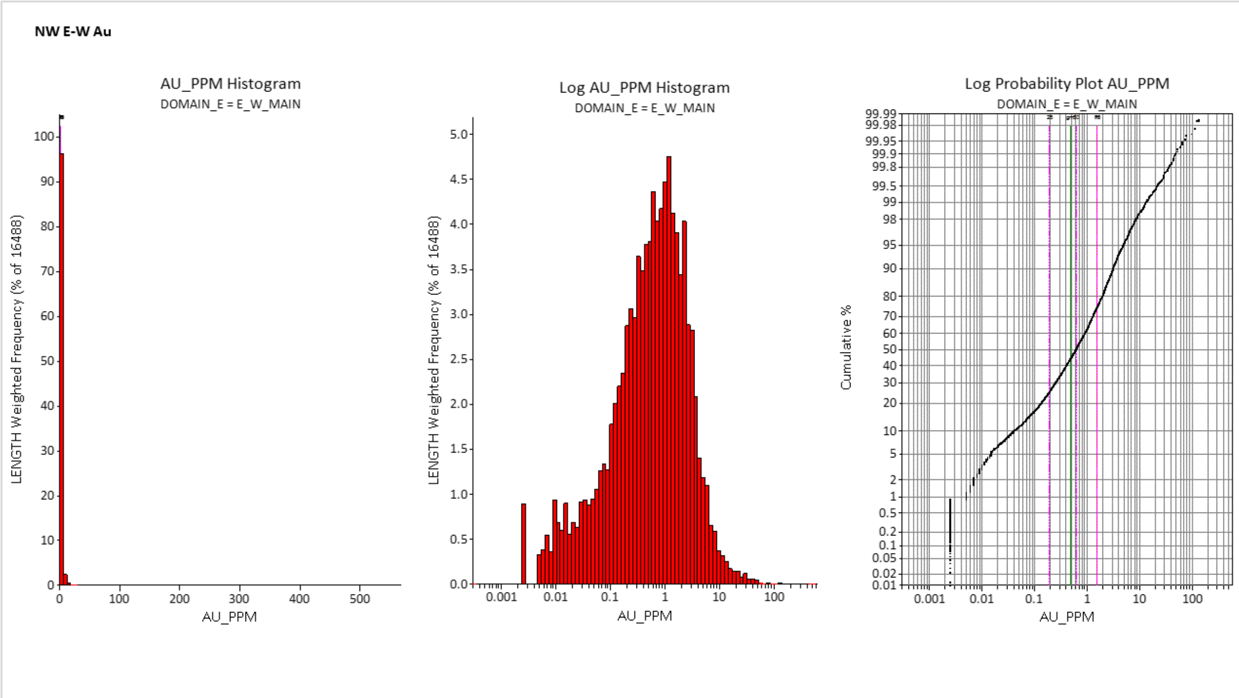


Figure 14-10: Gold histogram and probability plots for the Toroparu Deposit NW Area, E-W Domain

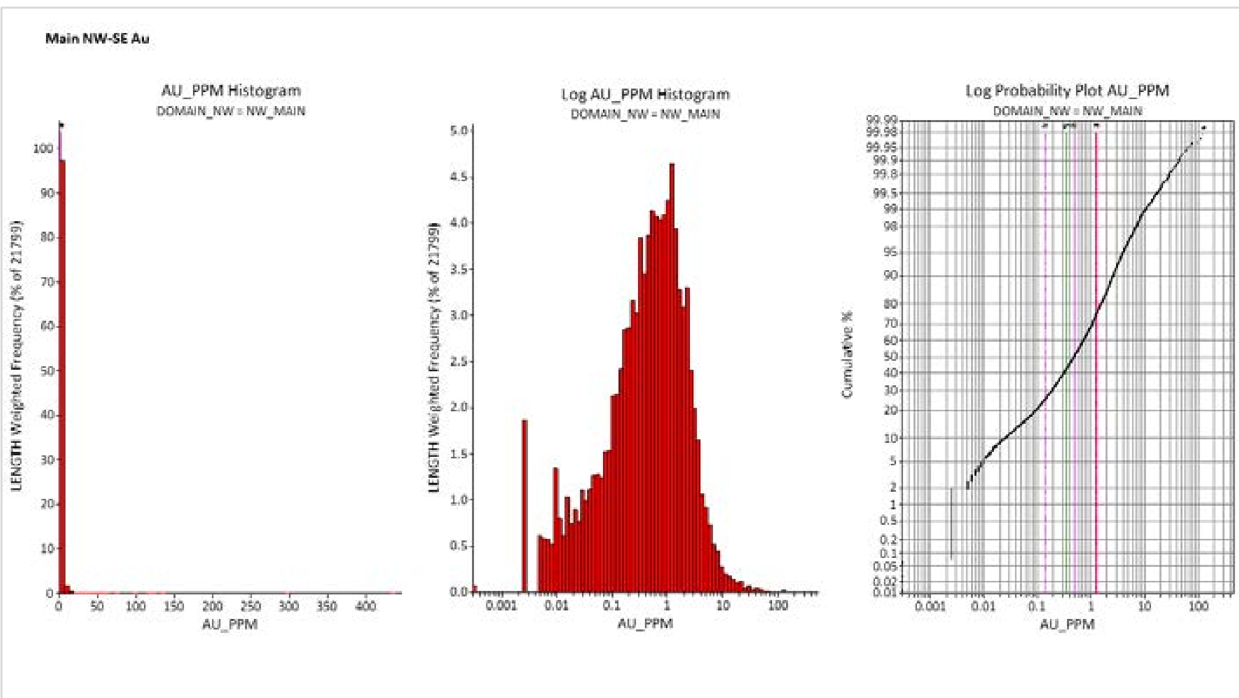


Figure 14-11: Gold histogram and probability plots for the Toroparu Deposit Main Area, NW-SE Domain

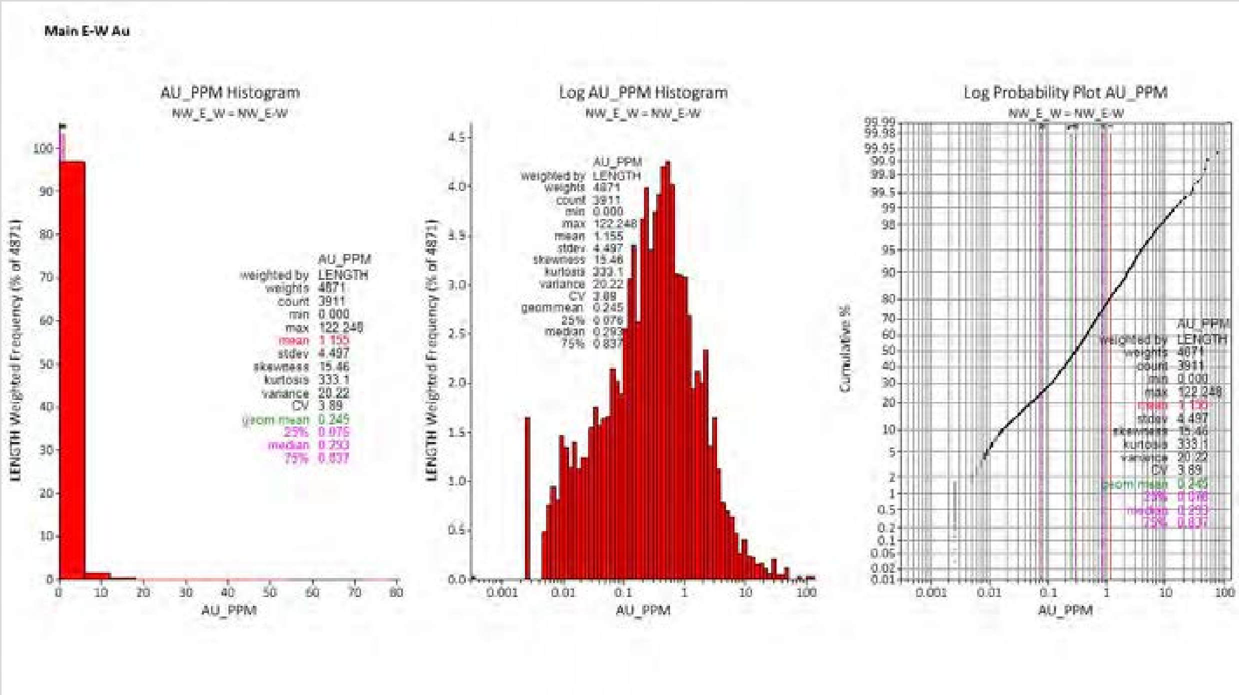


Figure 14-12: Gold histogram and probability plots for the Toroparu Deposit Main Area, E-W Domain

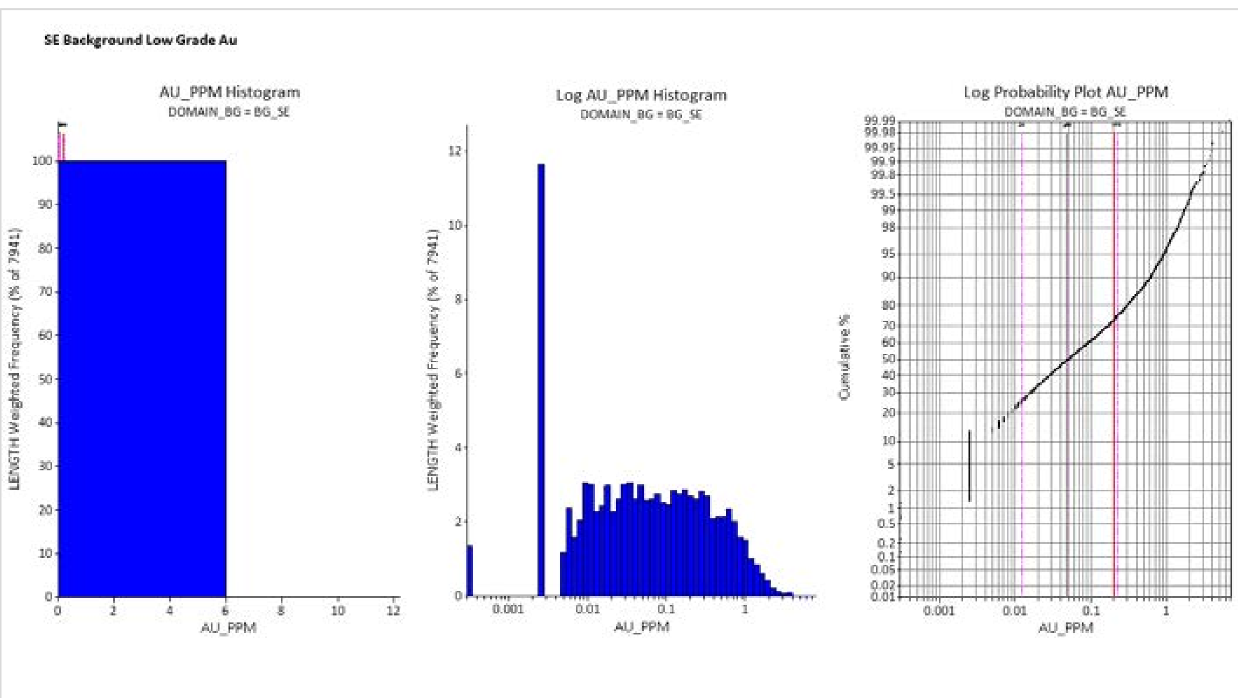


Figure 14-13: Gold histogram and probability plots for the Toroparu Deposit SE Area, Background Low-Grade Domain

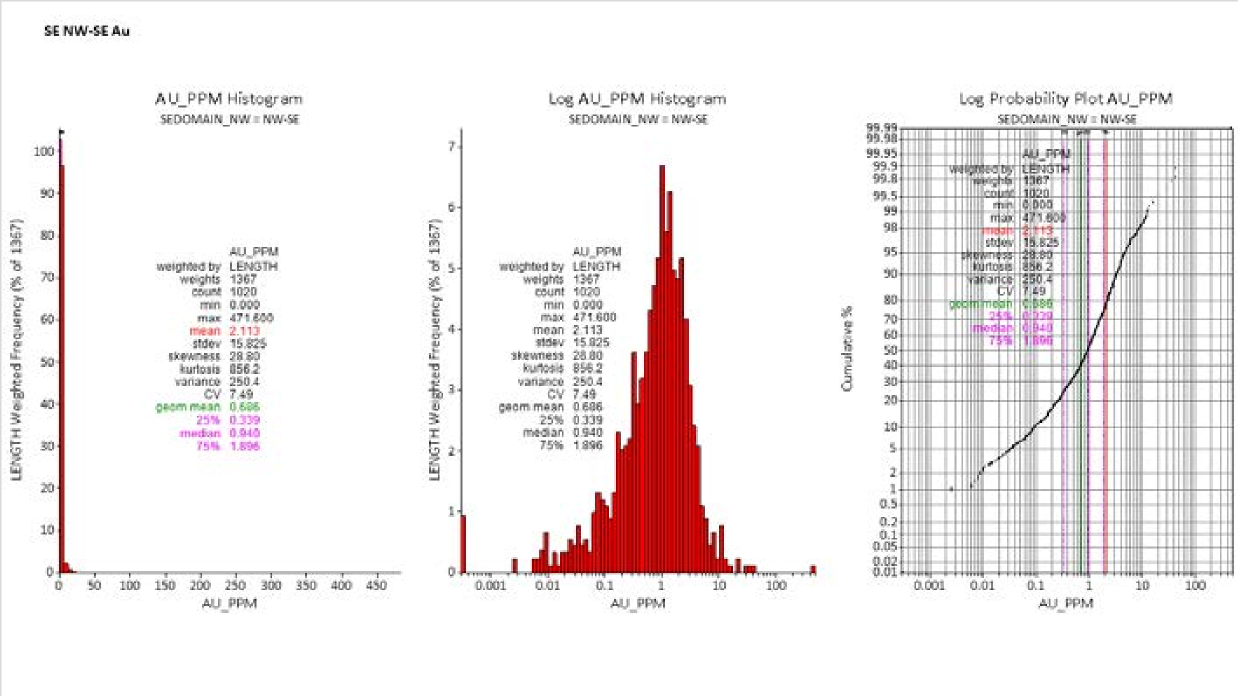


Figure 14-14: Gold histogram and probability plots for the Toroparu Deposit SE Area, NW-SE Domain

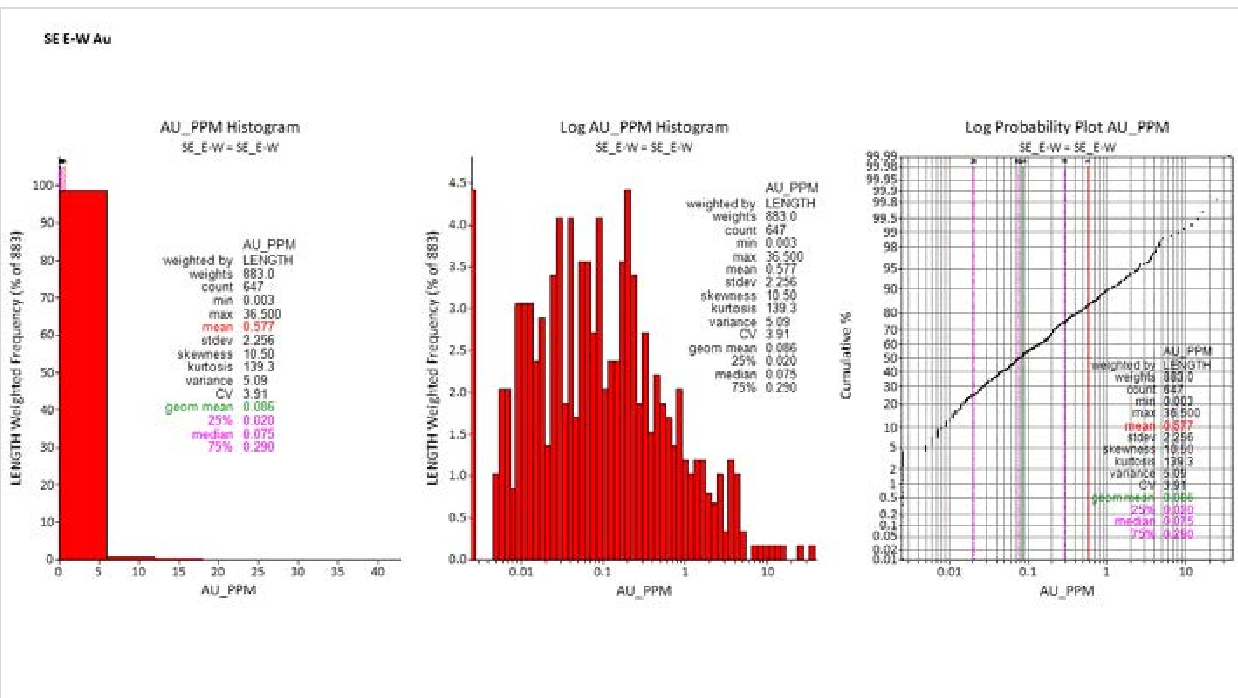


Figure 14-15: Gold histogram and probability plots for the Toroparu Deposit SE Area, E-W Domain

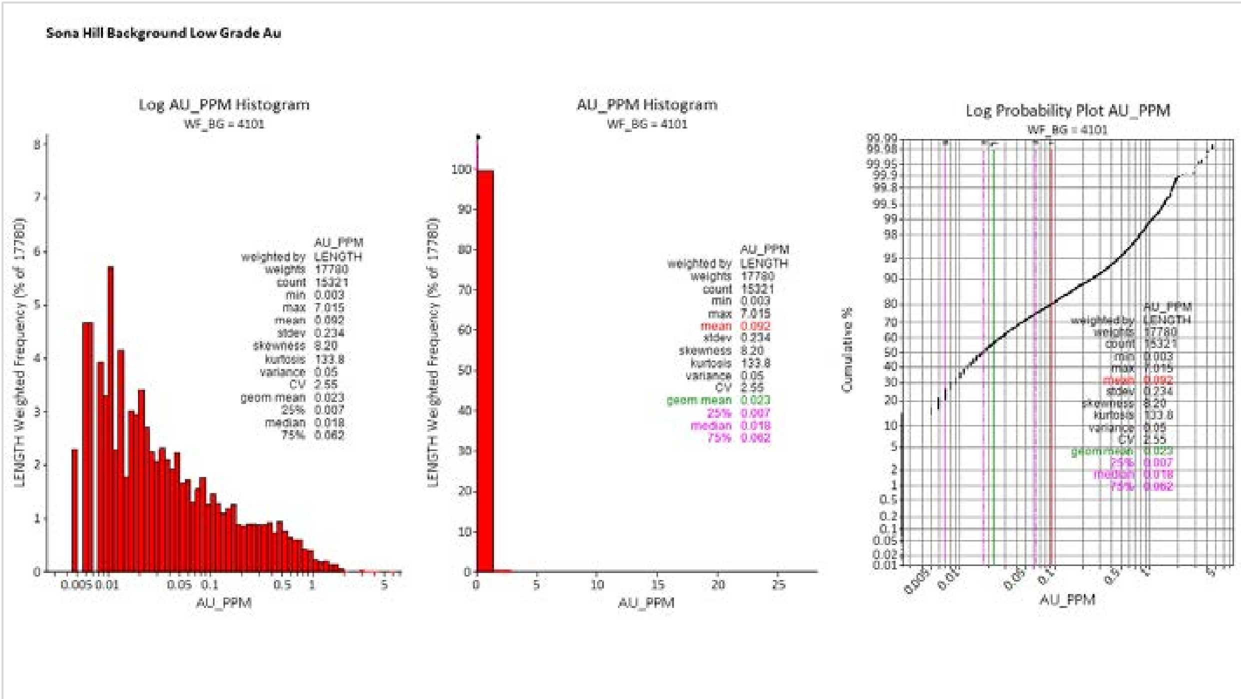


Figure 14-16: Gold histogram and probability plots for the Sona Hill Deposit, Background Low-Grade Domain

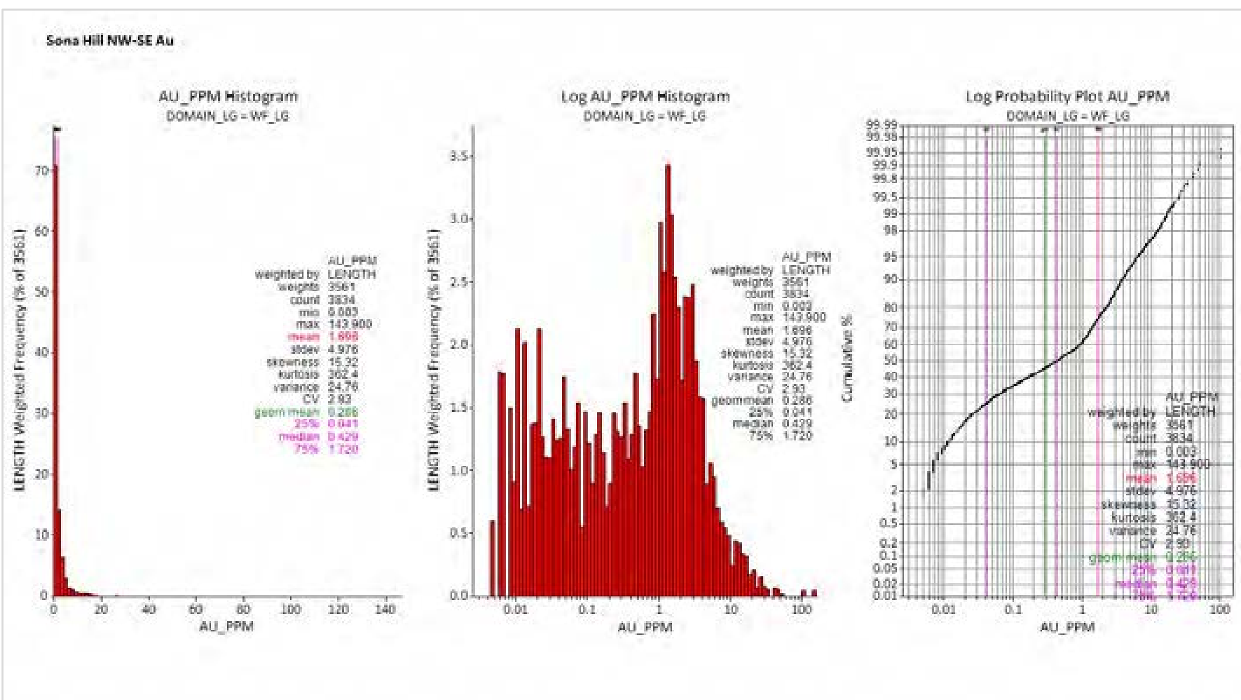


Figure 14-17: Gold histogram and probability plots for the Sona Hill Deposit, NW-SE Domain

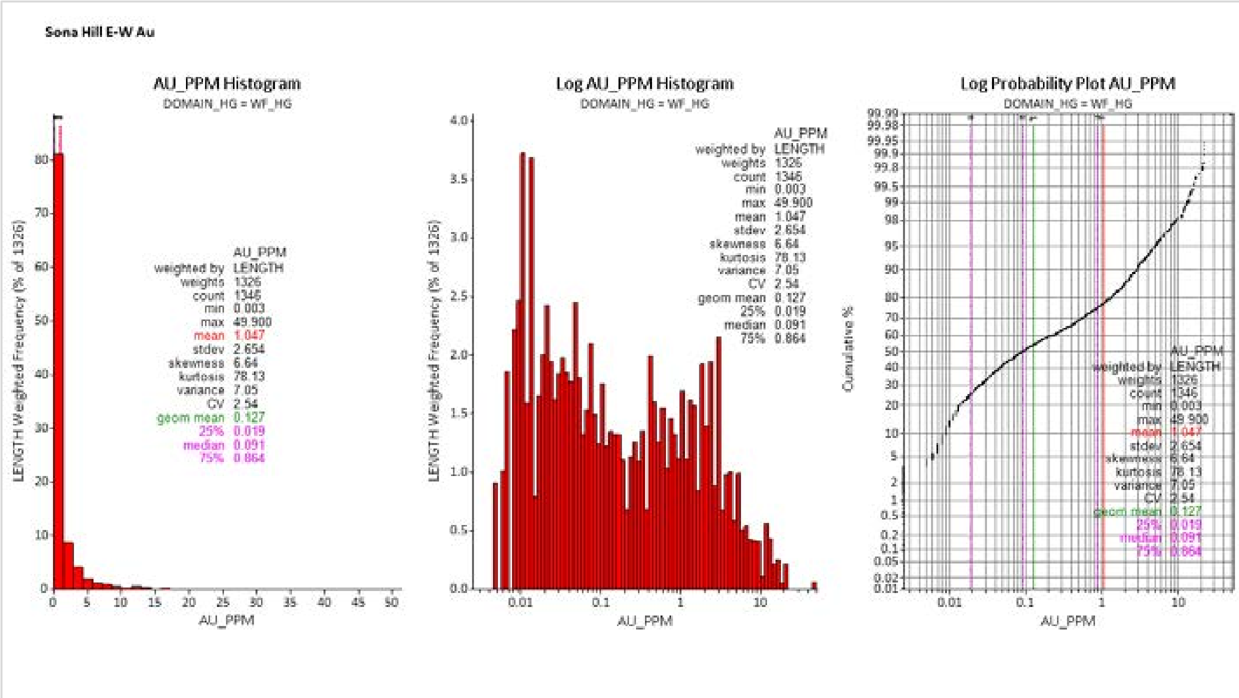


Figure 14-18: Gold Histogram and probability plots for the Sona Hill Deposit, E-W Domain

14.4 Data Preparation

Prior to grade estimation, the data was prepared in the following manner for both the Toroparu and Sona Hill Deposit:

- Very few unsampled intervals exist, but any that did were assigned a half-detection grade for gold, copper, and silver.
- The raw assay data was manually coded (“flagged”) to Background Low-Grade, NW-SE, and E-W Domain wireframes by assigning codes representative to the deposit, area, domain, and wireframe.
- Each wireframe’s flagged assays were statistically analyzed to define appropriate capping, modelling procedures, and parameters.
- High-grade outlier samples in each Domain were top-cut ("capped") to a variable maximum value.

Non Sampled Intervals and Minimum Detection Limits

Table 14-7 summarizes the drill hole assays at minimum detection used in the resource model. The assay table received by Nordmin contained half-minimum detection gold values substituted for assays below minimum detection. In addition, when non assayed gold intervals exist for payable and non payable fields, half-minimum detection values were substituted to remove bias from the block model. Values in

Table 14-7 are based on the master drill hole file defined in Section 14.3.

Table 14-7: Assays at Minimum Detection, Toroparu Deposit

Field	Count	Minimum Detection Limit	Count at Minimum Detection	% at Minimum Detection
Au (g/t)	136,938	0.0005	391	0.29%
		Total:	17,595	12.85%
Ag (g/t)	27,949	0.001	21	0.08%
		Total:	5,329	19.07%
Cu (g/t)	128,536	0.005	13	0.01%

Table 14-8: Assays at Minimum Detection, Sona Hill Deposit

Field	Count	Minimum Detection Limit	Count at Minimum Detection	% at Minimum Detection
Au (g/t)	16,005	0.005	2,501	15.63%
Ag (g/t)	2,711	0.001	1	0.00%
Cu (g/t)	0	n/a	0	0.00%

14.4.1 Outlier Analysis and Capping

Grade outliers are high-grade assay values that are much higher than the general population of samples and have the potential to bias (inflate) the quantity of metal estimated in a block model. Geostatistical analysis by Nordmin employed XY scatter plots, cumulative probability plots, and decile analysis to analyze the raw drill hole assay data for each wireframe to determine appropriate grade capping. The X10 Geo software package performed the statistical analysis. Table 14-9, Table 14-10, Table 14-11 are cap summaries for all wireframes; Table 14-12,

Table 14-13, and Table 14-14 are the results from the gold capping analysis for all wireframes.

Each domain wireframe flagged raw assays were statistically analyzed to define appropriate capping, modelling procedures, and parameters. Assays were variably capped to prevent excessive high-grade from skewing the estimation in each wireframe.

Table 14-9: Cap Values, Toroparu Main, and NW Areas

		Background Wireframes				NW-SE Wireframes				E-W Wireframes			
Deposit	Area	Wireframe	Au Cap	Cu Cap	Ag Cap	Wireframe	Au Cap	Cu Cap	Ag Cap	Wireframe	Au Cap	Cu Cap	Ag Cap
Toroparu		Main/NW	15.0	2.5	60.0								
Toroparu	Main	The NW-SE and E-W wireframes within the Main Area are encompassed by the Main/NW background wireframe				1	90.0	No cap	No cap	1	60.0	No cap	No cap
						2	No cap	No cap	No cap	2	No cap	1.0	No cap
						3	90.0	No cap	No cap	3	90.0	No cap	No cap
						4	60.0	3.0	60.0	4	No cap	No cap	No cap
						5	45.0	No cap	No cap	5	No cap	3.0	60.0
						6	90.0	3.0	60.0	6	30.0	No cap	No cap
						7	90.0	No cap	No cap	7	90.0	No cap	No cap
						8	30.0	No cap	45.0	8	90.0	1.0	No cap
						9	No cap	No cap	No cap	9	No cap	No cap	No cap
										10	30.0	No cap	No cap
										11	No cap	No cap	30.0
										12	No cap	No cap	No cap
Toroparu	NW	The NW-SE and E-W wireframes within the NW Area are encompassed by the Main/NW background wireframe				1	No cap	No cap	No cap	1	20.0	No cap	No cap
						2	No cap	0.8	No cap	2	No cap	No cap	No cap
						3	No cap	No cap	No cap	3	20.0	No cap	No cap
						4	No cap	No cap	No cap	4	No cap	No cap	No cap
						5	20.0	0.4	No cap	5	No cap	0.8	No cap
						6	90.0	0.5	No cap	6	90.0	1.0	15.0
						7	30.0	0.5	No cap	7	30.0	No cap	No cap
						8	90.0	1.0	15.0	8	No cap	1.2	No cap
						9	30.0	1.0	No cap	9	40.0	1.2	No cap
						10	30.0	1.5	No cap	10	40.0	0.8	12.0
						12	No cap	No cap	No cap	11	60.0	0.2	No cap
										12	No cap	No cap	No cap
										13	No cap	No cap	No cap
										14	40.0	1.2	No cap
										15	90.0	No cap	No cap
										16	30.0	No cap	No cap

Table 14-10: Cap Values, Toroparu SE, SEHW, and SEFW Areas

		Background Wireframes				NW-SE Wireframes				E-W Wireframes			
Deposit	**Area**	**Wireframe**	**Au Cap**	**Cu Cap**	**Domain**	**Wireframe**	**Au Cap**	**Cu Cap**	**Ag Cap**	**Wireframe**	**Au Cap**	**Cu Cap**	**Ag Cap**
Toroparu	SE/SEHW/SEFW	Background	No cap	1.0	No cap								
Toroparu	SE	The NW-SE and E-W wireframes within the SE Area are encompassed by the SE/SEHW/SEFW background wireframe				2	No cap	No cap	No cap	1	5.0	No cap	No cap
						3	23.0	No cap	No cap	2	20.0	1.0	No cap
						4	No cap	No cap	No cap	3	No cap	No cap	No cap
						5	No cap	No cap	No cap	4	20.0	1.5	No cap
						6	No cap	No cap	No cap	5	5.0	No cap	No cap
						7	No cap	No cap	No cap				
						8	No cap	No cap	No cap				
						9	No cap	No cap	No cap				
						10	No cap	No cap	No cap				
						11	No cap	No cap	No cap				
						13	No cap	No cap	No cap				
						14	No cap	No cap	No cap				
						15	No cap	No cap	No cap				
						16	No cap	No cap	No cap				
						17	No cap	No cap	No cap				
						18	No cap	No cap	No cap				
						19	No cap	No cap	No cap				
						20	No cap	No cap	No cap				
						21	No cap	No cap	No cap				
						22	No cap	No cap	No cap				
						23	No cap	No cap	No cap				
						24	No cap	No cap	No cap				
						25	No cap	No cap	No cap				
						26	No cap	No cap	No cap				
Toroparu	SEHW	The NW-SE wireframes for the SEHW Area are encompassed by the SE/SEHW/SEFW background wireframe				2	No cap	No cap	No cap	No E-W wireframes exist in the SEHW Area			
						3	No cap	No cap	No cap				
						4	No cap	No cap	No cap				
						5	No cap	No cap	No cap				
						6	No cap	No cap	No cap				
						7	No cap	No cap	No cap				
						8	No cap	No cap	No cap				
						9	No cap	No cap	No cap				

<table>
<tr><th></th><th></th><th colspan="4">Background Wireframes</th><th colspan="4">NW-SE Wireframes</th><th colspan="4">E-W Wireframes</th></tr>
<tr><th>Deposit</th><th>Area</th><th>Wireframe</th><th>Au Cap</th><th>Cu Cap</th><th>Domain</th><th>Wireframe</th><th>Au Cap</th><th>Cu Cap</th><th>Ag Cap</th><th>Wireframe</th><th>Au Cap</th><th>Cu Cap</th><th>Ag Cap</th></tr>
<tr><td rowspan="4"></td><td rowspan="4"></td><td rowspan="4" colspan="4"></td><td>10</td><td>No cap</td><td>No cap</td><td>No cap</td><td rowspan="4" colspan="4"></td></tr>
<tr><td>11</td><td>No cap</td><td>No cap</td><td>No cap</td></tr>
<tr><td>12</td><td>No cap</td><td>No cap</td><td>No cap</td></tr>
<tr><td>13</td><td>No cap</td><td>No cap</td><td>No cap</td></tr>
<tr><td rowspan="11">Toroparu</td><td rowspan="11">SEFW</td><td rowspan="11" colspan="4">The NW-SE wireframes for the SEFW Area are encompassed by the SE/SEHW/SEFW background wireframe</td><td>2</td><td>20.0</td><td>No cap</td><td>No cap</td><td rowspan="11" colspan="4">No E-W wireframes exist in the SEFW Area</td></tr>
<tr><td>3</td><td>No cap</td><td>No cap</td><td>No cap</td></tr>
<tr><td>4</td><td>30.0</td><td>No cap</td><td>No cap</td></tr>
<tr><td>6</td><td>No cap</td><td>0.3</td><td>No cap</td></tr>
<tr><td>7</td><td>No cap</td><td>No cap</td><td>No cap</td></tr>
<tr><td>8</td><td>No cap</td><td>No cap</td><td>No cap</td></tr>
<tr><td>9</td><td>No cap</td><td>No cap</td><td>No cap</td></tr>
<tr><td>10</td><td>No cap</td><td>No cap</td><td>No cap</td></tr>
<tr><td>11</td><td>No cap</td><td>No cap</td><td>No cap</td></tr>
<tr><td>12</td><td>No cap</td><td>No cap</td><td>No cap</td></tr>
<tr><td>13</td><td>No cap</td><td>No cap</td><td>No cap</td></tr>
</table>

Table 14-11: Cap Values, Sona Hill Deposit

		Background Wireframes				NW-SE Wireframes				E-W Wireframes			
Deposit	**Area**	**Wireframe**	**Au Cap**	**Cu Cap**	**Domain**	**Wireframe**	**Au Cap**	**Cu Cap**	**Ag Cap**	**Wireframe**	**Au Cap**	**Cu Cap**	**Ag Cap**
Toroparu	SE/SEHW/ SEFW	Background	2.0	No cap	3.5								
Sona Hill	n/a	The Sona Hill NW-SE wireframes are encompassed by the Sona Hill background wireframe				1	14.0	14.0	10.0	1	No Cap	No Cap	No Cap
						2	30.0	30.0	7.0	2	No Cap	No Cap	5.0
						3	30.0	30.0	10.0	3	No Cap	No Cap	No Cap
						4	30.0	30.0	15.0	4	No Cap	No Cap	No Cap
						5	30.0	30.0	5.0	5	6.0	No Cap	No Cap
						6	No Cap	No Cap	No Cap	6	8.0	No Cap	No Cap
						7	No Cap	No Cap	No Cap	7	No Cap	No Cap	15.0
						8	15.0	30.0	No Cap	8	15.0	No Cap	No Cap
						9	5.0	5.0	No Cap	9	30.0	No Cap	15.0
						10	15.0	15.0	6.0	10	No Cap	No Cap	No Cap
						11	8.0	8.0	2.5	11	No Cap	No Cap	5.0
						12	5.0	5.0	No Cap				
						13	No Cap	No Cap	No Cap				
						14	10.0	10.0	No Cap				

Table 14-12: Toroparu Main and NW Area Gold Outlier Analysis and Capping

							Capped							Uncapped			
Deposit	Area	Domain	Wireframe	Metal	Cap (g/t)	# of Samples	Min	Max	Mean	# Capped	% Capped	% Metal Lost	CV	Min	Max	Mean	CV
Toroparu	Main/NW	Background	1	Au	15.0	92,519	0.0010	15.00	0.17	4	0.0	0.0	0.92	0.0010	19.20	0.17	1.91
	Main	NW-SE	1	Au	90.0	688	0.0025	90.00	0.41	1	0.1	45.0	9.02	0.0025	298.90	0.74	16.17
			2	Au	No cap	180	0.0025	11.50	0.62	0	0.0	0.0	2.35	0.0025	11.50	0.62	2.35
			3	Au	90.0	1,300	0.0030	90.00	1.09	4	0.3	16.0	3.05	0.0030	135.00	1.13	3.93
			4	Au	60.0	1,121	0.0030	60.00	1.01	1	0.1	0.1	2.31	0.0030	64.40	1.02	2.34
			5	Au	45.0	2,267	0.0030	45.00	1.35	2	0.1	0.8	2.13	0.0030	59.00	1.36	2.24
			6	Au	90.0	3,954	0.0030	90.00	1.52	1	0.0	0.1	1.90	0.0030	98.40	1.52	1.92
			7	Au	90.0	2,106	0.0025	90.00	1.15	1	0.0	5.2	2.14	0.0025	432.00	1.21	5.14
			8	Au	30.0	1,778	0.0025	30.00	0.97	2	0.1	0.9	2.08	0.0025	38.54	0.98	2.20
			9	Au	No cap	399	0.0025	11.06	1.15	0	0.0	0.0	1.38	0.0025	11.06	1.15	1.38
		E-W	1	Au	60.0	606	0.0030	60.00	1.45	2	0.3	0.2	2.84	0.0030	64.40	1.46	2.87
			2	Au	No cap	550	0.0030	36.70	1.26	0	0.0	0.0	2.08	0.0030	36.70	1.26	2.08
			3	Au	90.0	671	0.0025	90.00	2.08	2	0.3	8.8	2.72	0.0025	432.00	2.28	5.09
			4	Au	No cap	979	0.0030	26.84	1.67	0	0.0	0.0	1.41	0.0030	26.84	1.67	1.41
			5	Au	No cap	2,074	0.0030	64.20	1.69	0	0.0	0.0	1.81	0.0030	64.20	1.69	1.81
			6	Au	30.0	492	0.0030	30.00	1.63	1	0.2	0.4	1.74	0.0030	38.40	1.63	1.78
			7	Au	90.0	1,023	0.0030	90.00	1.81	4	0.4	2.7	2.23	0.0030	135.00	1.85	2.82
			8	Au	90.0	573	0.0030	90.00	1.63	2	0.3	17.0	1.47	0.0030	555.10	1.98	7.77
			9	Au	No cap	339	0.0050	29.00	1.50	0	0.0	0.0	1.85	0.0050	29.00	1.50	1.85
			10	Au	30.0	659	0.0030	30.00	1.24	1	0.2	1.2	1.79	0.0030	38.54	1.25	1.94
			11	Au	No cap	565	0.0030	23.20	1.29	0	0.0	0.0	1.58	0.0030	23.20	1.29	1.58
			12	Au	No cap	472	0.0030	35.40	1.55	0	0.0	0.0	2.33	0.0030	35.40	1.55	2.33
	NW	NW-SE	1	Au	No cap	12	0.1040	10.89	2.18	0	0.0	0.0	1.54	0.1040	10.89	2.18	1.54
			2	Au	No cap	62	0.0030	15.55	1.23	0	0.0	0.0	1.96	0.0030	15.55	1.23	1.96
			3	Au	No cap	115	0.0130	21.60	1.24	0	0.0	0.0	2.04	0.0130	21.60	1.24	2.04
			4	Au	No cap	214	0.0030	19.60	1.06	0	0.0	0.0	2.10	0.0030	19.60	1.06	2.10
			5	Au	20.0	285	0.0060	20.00	1.06	2	0.7	5.4	2.25	0.0060	29.50	1.12	2.60
			6	Au	90.0	367	0.0030	90.00	1.37	1	0.3	7.3	4.54	0.0030	122.20	1.48	5.30
			7	Au	30.0	709	0.0030	30.00	1.04	4	0.6	5.6	3.06	0.0030	48.60	1.10	3.48

							Capped							Uncapped			
Deposit	**Area**	**Domain**	**Wireframe**	**Metal**	**Cap (g/t)**	**# of Samples**	**Min**	**Max**	**Mean**	**# Capped**	**% Capped**	**% Metal Lost**	**CV**	**Min**	**Max**	**Mean**	**CV**
			8	Au	90.0	497	0.0030	90.00	1.40	1	0.2	5.0	4.63	0.0030	120.80	1.50	5.14
			9	Au	30.0	528	0.0030	30.00	0.95	1	0.2	3.4	2.97	0.0030	43.90	1.00	3.27
			10	Au	30.0	354	0.0030	30.00	0.78	1	0.3	3.3	3.11	0.0030	37.50	0.81	3.43
			12	Au	No cap	1	16.5900	16.59	16.59	0	0.0	0.0	0.00	16.5900	16.59	16.59	0.00
		E-W	1	Au	20.0	293	0.0030	20.00	0.79	1	0.3	1.1	2.71	0.0030	22.30	0.80	2.79
			2	Au	No cap	51	0.0070	15.90	1.32	0	0.0	0.0	2.27	0.0070	15.90	1.32	2.27
			3	Au	20.0	247	0.0030	26.50	0.74	0	0.0	0.0	3.00	0.0030	26.50	0.74	3.00
			4	Au	No cap	25	0.0030	0.10	1.80	0	0.0	0.0	1.41	0.0030	0.10	1.80	1.41
			5	Au	No cap	78	0.0030	4.42	0.87	0	0.0	0.0	1.24	0.0030	4.42	0.87	1.24
			6	Au	90.0	529	0.0030	90.00	1.22	1	0.2	5.7	3.89	0.0030	122.20	1.30	4.77
			7	Au	30.0	582	0.0030	30.00	1.09	1	0.2	2.8	2.69	0.0030	43.90	1.12	2.94
			8	Au	No cap	338	0.0030	15.30	0.86	0	0.0	0.0	1.76	0.0030	15.30	0.86	1.76
			9	Au	40.0	569	0.0030	40.00	0.96	1	0.2	1.4	3.28	0.0030	46.20	0.97	3.42
			10	Au	40.0	387	0.0010	40.00	1.10	2	0.5	3.7	3.69	0.0010	48.60	1.15	3.92
			11	Au	60.0	127	0.0030	60.00	1.64	1	0.8	8.4	3.90	0.0030	76.60	1.80	4.36
			12	Au	No cap	87	0.0030	17.50	0.94	0	0.0	0.0	1.77	0.0030	17.50	0.94	1.77
			13	Au	No cap	174	0.0030	11.70	0.89	0	0.0	0.0	1.90	0.0030	11.70	0.89	1.90
			14	Au	40.0	230	0.0030	40.00	1.31	1	0.4	4.6	2.97	0.0030	52.00	1.40	3.32
			15	Au	90.0	127	0.0030	90.00	2.90	1	0.8	7.1	3.41	0.0030	114.20	3.20	3.80
			16	Au	30.0	67	0.0050	30.00	1.43	1	1.5	7.5	2.57	0.0050	40.50	1.54	3.00

Table 14-13: Toroparu SE, SEHW, and SEFW Area Gold Outlier Analysis and Capping

							Capped							Uncapped			
Deposit	Area	Domain	Wireframe	Metal	Cap (g/t)	# of Samples	Min	Max	Mean	# Capped	% Capped	% Metal Lost	CV	Min	Max	Mean	CV
Toroparu	SE/SEHW/SEFW	Background	1	Au	No cap	5,974	0.0001	6.77	0.17	0	0.0	0.0	1.87	0.0001	6.77	0.17	1.87
	SE	NW-SE	2	Au	No cap	5	0.2610	11.90	3.95	0	0.0	0.0	1.15	0.2610	11.90	3.95	1.15
			3	Au	23.0	9	0.0230	24.80	6.11	0	0.0	0.0	1.53	0.0230	24.80	6.11	1.53
			4	Au	No cap	3	0.2920	14.20	5.14	0	0.0	0.0	1.53	0.2920	14.20	5.14	1.53
			5	Au	No cap	2	0.0080	16.00	6.86	0	0.0	0.0	1.65	0.0080	16.00	6.86	1.65
			6	Au	No cap	4	0.1600	3.72	1.89	0	0.0	0.0	1.05	0.1600	3.72	1.89	1.05
			7	Au	No cap	4	4.6020	4.60	4.60	0	0.0	0.0	0.00	4.6020	4.60	4.60	0.00
			8	Au	No cap	3	0.0310	2.37	1.09	0	0.0	0.0	1.09	0.0310	2.37	1.09	1.09
			9	Au	No cap	4	0.0060	2.21	0.62	0	0.0	0.0	1.70	0.0060	2.21	0.62	1.70
			10	Au	No cap	3	0.2620	2.20	0.94	0	0.0	0.0	1.16	0.2620	2.20	0.94	1.16
			11	Au	No cap	4	0.3490	4.14	1.42	0	0.0	0.0	1.29	0.3490	4.14	1.42	1.29
			13	Au	No cap	7	0.0320	5.81	1.32	0	0.0	0.0	1.59	0.0320	5.81	1.32	1.59
			14	Au	No cap	3	0.2550	2.17	1.06	0	0.0	0.0	0.94	0.2550	2.17	1.06	0.94
			15	Au	No cap	4	0.3510	6.03	2.56	0	0.0	0.0	0.99	0.3510	6.03	2.56	0.99
			16	Au	No cap	4	0.1220	2.29	0.78	0	0.0	0.0	1.31	0.1220	2.29	0.78	1.31
			17	Au	No cap	2	0.2190	2.59	1.41	0	0.0	0.0	1.19	0.2190	2.59	1.41	1.19
			18	Au	No cap	2	0.8600	2.12	1.49	0	0.0	0.0	0.60	0.8600	2.12	1.49	0.60
			19	Au	No cap	3	0.3000	2.60	1.28	0	0.0	0.0	0.93	0.3000	2.60	1.28	0.93
			20	Au	No cap	2	0.2470	6.55	3.39	0	0.0	0.0	1.31	0.2470	6.55	3.39	1.31
			21	Au	No cap	7	0.0810	4.23	3.03	0	0.0	0.0	0.65	0.0810	4.23	3.03	0.65
			22	Au	No cap	2	0.0570	2.25	1.15	0	0.0	0.0	1.34	0.0570	2.25	1.15	1.34

							Capped							Uncapped			
Deposit	Area	Domain	Wireframe	Metal	Cap (g/t)	# of Samples	Min	Max	Mean	# Capped	% Capped	% Metal Lost	CV	Min	Max	Mean	CV
			23	Au	No cap	2	1.9230	2.13	2.03	0	0.0	0.0	0.07	1.9230	2.13	2.03	0.07
			24	Au	No cap	3	0.3580	3.70	1.65	0	0.0	0.0	1.09	0.3580	3.70	1.65	1.09
			25	Au	No cap	5	0.3280	3.42	1.14	0	0.0	0.0	1.13	0.3280	3.42	1.14	1.13
			26	Au	No cap	3	0.9590	3.11	2.19	0	0.0	0.0	0.51	0.9590	3.11	2.19	0.51
		E-W	1	Au	5.0	97	0.0030	5.00	0.32	1	0.0	19.0	2.91	0.0030	9.50	0.37	3.40
			2	Au	20.0	243	0.0030	5.00	0.32	1	0.9	19.0	2.91	0.0030	9.50	0.37	3.40
			3	Au	No cap	38	0.0030	20.00	0.47	1	0.4	4.3	3.69	0.0030	24.80	0.50	4.04
			4	Au	20.0	137	0.0060	4.66	0.87	0	0.0	0.0	1.57	0.0060	4.66	0.87	1.57
			5	Au	5.0	132	0.0030	20.00	0.72	1	0.7	16.0	3.64	0.0030	36.50	0.85	4.42
	SEHW	NW-SE	2	Au	No cap	82	0.0010	10.80	1.36	0	0.0	0.0	1.44	0.0010	10.80	1.36	1.44
			3	Au	No cap	7	1.9400	12.80	6.02	0	0.0	0.0	0.83	1.9400	12.80	6.02	0.83
			4	Au	No cap	157	0.0100	30.00	1.84	1	0.6	2.3	2.14	0.0100	35.60	1.88	2.27
			5	Au	No cap	15	0.2240	10.60	1.77	0	0.0	0.0	1.57	0.2240	10.60	1.77	1.57
			6	Au	No cap	8	0.1880	9.22	2.50	0	0.0	0.0	1.51	0.1880	9.22	2.50	1.51
			7	Au	No cap	7	0.1480	3.51	1.31	0	0.0	0.0	0.99	0.1480	3.51	1.31	0.99
			8	Au	No cap	1	3.8720	3.87	3.87	0	0.0	0.0	0.00	3.8720	3.87	3.87	0.00
			9	Au	No cap	3	0.1660	2.10	1.11	0	0.0	0.0	0.87	0.1660	2.10	1.11	0.87
			10	Au	No cap	4	0.1900	8.08	2.92	0	0.0	0.0	1.37	0.1900	8.08	2.92	1.37
			11	Au	No cap	82	0.0010	10.80	1.36	0	0.0	0.0	1.44	0.0010	10.80	1.36	1.44
			12	Au	No cap	7	1.9400	12.80	6.02	0	0.0	0.0	0.83	1.9400	12.80	6.02	0.83
			13	Au	No cap	157	0.0100	30.00	1.84	1	0.6	2.3	2.14	0.0100	35.60	1.88	2.27
	SEFW	NW-SE	2	Au	20.0	8	0.0370	20.00	3.96	2	0.3	48.0	1.97	0.0370	41.80	7.59	2.18
			3	Au	No cap	35	0.0100	8.70	0.75	0	0.0	0.0	2.26	0.0100	8.70	0.75	2.26

							Capped							Uncapped			
Deposit	Area	Domain	Wireframe	Metal	Cap (g/t)	# of Samples	Min	Max	Mean	# Capped	% Capped	% Metal Lost	CV	Min	Max	Mean	CV
			4	Au	30.0	4	0.4500	30.00	8.34	1	25.0	93.0	1.73	0.4500	471.60	118.70	1.98
			6	Au	No cap	254	0.0030	29.40	1.64	0	0.0	0.0	1.57	0.0070	7.20	1.36	0.88
			7	Au	No cap	375	0.0070	7.20	1.36	0	0.0	0.0	0.88	0.0820	2.80	1.30	0.94
			8	Au	No cap	4	0.0820	2.80	1.30	0	0.0	0.0	0.94	0.0110	3.98	1.53	0.88
			9	Au	No cap	12	0.0110	3.98	1.53	0	0.0	0.0	0.88	0.0630	2.09	1.07	1.33
			10	Au	No cap	2	0.0630	2.09	1.07	0	0.0	0.0	1.33	0.0080	8.44	3.07	1.52
			11	Au	No cap	3	0.0080	8.44	3.07	0	0.0	0.0	1.52	0.1070	3.14	1.62	1.32
			12	Au	No cap	2	0.1070	3.14	1.62	0	0.0	0.0	1.32	0.1470	2.91	1.10	1.12
			13	Au	No cap	7	0.1470	2.91	1.10	0	0.0	0.0	1.12	0.1470	2.91	1.10	1.12

Table 14-14: Sona Hill Deposit Gold Outlier Analysis and Capping

						Capped							Uncapped			
Deposit	Domain	Wireframe	Metal	Cap (g/t)	# of Samples	Min	Max	Mean	# Capped	% Capped	% Metal Lost	CV	Min	Max	Mean	CV
Sona Hill	Background	1	Au	2.0	15,321	0.0030	2.00	0.09	17	0.1	2.0	2.30	0.0030	7.02	0.09	2.55
	NW-SE	1	Au	14.0	523	0.0030	14.00	1.52	13	0.0	2.5	1.67	0.0030	18.60	1.56	1.76
		2	Au	30.0	798	0.0030	30.00	1.49	4	0.5	20.0	2.19	0.0030	143.90	1.86	4.43
		3	Au	30.0	982	0.0030	30.00	1.66	6	0.0	2.2	2.26	0.0030	50.40	1.69	2.40
		4	Au	30.0	359	0.0030	30.00	1.33	4	1.1	3.7	2.39	0.0030	49.90	1.38	2.70
		5	Au	30.0	333	0.0030	30.00	1.81	1	0.0	2.3	2.01	0.0030	41.80	1.85	2.17
		6	Au	No cap	15	0.1550	4.46	1.71	0	0.0	0.0	0.82	0.1550	4.46	1.71	0.82
		7	Au	No cap	25	0.0230	7.25	2.38	0	0.0	0.0	1.06	0.0230	7.25	2.38	1.06
		8	Au	30.0	106	0.0030	30.00	1.34	3	2.8	16.0	3.03	0.0030	48.40	1.59	3.75
		9	Au	5.0	24	0.0030	5.00	1.27	2	0.1	41.0	1.40	0.0030	16.80	2.15	2.00
		10	Au	15.0	254	0.0030	15.00	1.87	3	1.2	2.3	1.66	0.0030	18.40	1.92	1.72
		11	Au	8.0	161	0.0050	8.00	1.02	4	0.0	4.5	1.70	0.0050	13.10	1.07	1.84
		12	Au	5.0	99	0.0070	5.00	1.63	14	14.1	27.0	1.02	0.0070	16.80	2.24	1.53
		13	Au	No cap	81	0.0070	5.89	1.29	0	0.0	0.0	1.13	0.0070	5.89	1.29	1.13
		14	Au	10.0	67	0.0030	10.00	1.18	3	4.5	14.0	2.05	0.0030	19.20	1.38	2.36
	E-W	1	Au	No cap	157	0.0030	15.70	0.79	0	0.0	0.0	2.54	15.7000	0.79	15.70	2.54
		2	Au	No cap	183	0.0030	22.10	0.98	0	0.0	0.0	2.61	22.1000	0.98	22.10	2.61
		3	Au	No cap	230	0.0030	16.40	1.01	0	0.0	0.0	2.34	16.4000	1.01	16.40	2.34
		4	Au	No cap	39	0.0060	4.52	0.83	0	0.0	0.0	1.49	4.5170	0.83	4.52	1.49
		5	Au	6.0	21	0.0030	6.00	0.65	2	0.1	11.0	2.13	0.0030	8.08	0.74	2.37
		6	Au	8.0	107	0.0050	8.00	0.80	9	8.4	6.2	2.11	0.0050	13.90	0.86	2.31
		7	Au	No cap	69	0.0030	6.52	0.57	0	0.0	0.0	2.14	0.0030	6.52	0.57	2.14

						Capped							Uncapped			
		8	Au	15.0	140	0.0030	15.00	1.08	3	2.1	7.1	2.68	0.0030	21.30	1.16	2.88
		9	Au	30.0	213	0.0030	30.00	1.63	1	0.0	4.5	2.14	0.0030	49.90	1.71	2.50
		10	Au	No cap	87	0.0050	11.60	1.19	0	0.0	0.0	2.08	0.0050	11.60	1.19	2.08
		11	Au	No cap	105	0.0030	9.03	0.92	0	0.0	0.0	1.59	0.0030	9.03	0.92	1.59

Table 14-15: Toroparu Main and NW Area Copper Outlier Analysis and Capping

							Capped							Uncapped			
Deposit	Area	Domain	Wireframe	Metal	Cap (g/t)	# of Samples	Min	Max	Mean	# Capped	% Capped	% Metal Lost	CV	Min	Max	Mean	CV
Toroparu	Main/NW	Background	1	Cu	2.5	89,234	0.0000	2.50	0.04	2	0.0	0.0	1.91	0.0000	3.42	0.04	1.91
	Main	NW-SE	1	Cu	No cap	687	0.0030	0.71	0.04	0	0.0	0.0	1.85	0.0030	0.71	0.04	1.85
			2	Cu	No cap	172	0.0030	0.67	0.10	0	0.0	0.0	1.18	0.0030	0.67	0.10	1.18
			3	Cu	No cap	1,295	0.0030	1.40	0.14	0	0.0	0.0	1.18	0.0030	1.40	0.14	1.18
			4	Cu	3.0	1,118	0.0030	3.00	0.12	1	0.1	0.1	1.22	0.0030	3.44	0.12	1.24
			5	Cu	No cap	2,231	0.0030	2.26	0.12	0	0.0	0.0	1.27	0.0030	2.26	0.12	1.27
			6	Cu	3.0	3,942	0.0030	3.00	0.19	1	0.0	0.3	1.24	0.0030	4.66	0.19	1.28
			7	Cu	No cap	2,084	0.0030	0.75	0.08	0	0.0	0.0	1.17	0.0030	0.75	0.08	1.17
			8	Cu	No cap	1,718	0.0030	1.73	0.06	0	0.0	0.0	1.35	0.0030	1.73	0.06	1.35
			9	Cu	No cap	376	0.0010	0.97	0.12	0	0.0	0.0	1.30	0.0010	0.97	0.12	1.30
		E-W	1	Cu	No cap	602	0.0010	0.95	0.10	0	0.0	0.0	1.32	0.0010	0.95	0.10	1.32
			2	Cu	1.0	532	0.0030	1.40	0.08	0	0.0	0.0	1.53	0.0030	1.40	0.08	1.53
			3	Cu	No cap	658	0.0030	1.80	0.15	0	0.0	0.0	1.00	0.0030	1.80	0.15	1.00
			4	Cu	No cap	979	0.0030	2.35	0.22	0	0.0	0.0	1.12	0.0030	2.35	0.22	1.12
			5	Cu	3.0	2,031	0.0030	3.00	0.22	1	0.1	0.4	1.10	0.0030	4.65	0.22	1.16
			6	Cu	No cap	485	0.0030	0.96	0.13	0	0.0	0.0	1.04	0.0030	0.96	0.13	1.04
			7	Cu	No cap	1,013	0.0030	2.26	0.10	0	0.0	0.0	1.35	0.0030	2.26	0.10	1.35
			8	Cu	1.0	573	0.0010	1.00	0.07	1	0.2	0.2	1.39	0.0010	1.06	0.07	1.40
			9	Cu	No cap	334	0.0030	0.79	0.09	0	0.0	0.0	1.37	0.0030	0.79	0.09	1.37
			10	Cu	No cap	656	0.0030	0.73	0.06	0	0.0	0.0	1.19	0.0030	0.73	0.06	1.19
			11	Cu	No cap	565	0.0010	1.73	0.06	0	0.0	0.0	1.67	0.0010	1.73	0.06	1.67
			12	Cu	No cap	472	0.0010	0.96	0.08	0	0.0	0.0	1.47	0.0010	0.96	0.08	1.47
	NW	NW-SE	1	Cu	No cap	12	0.0030	0.06	0.02	0	0.0	0.0	0.85	0.0030	0.06	0.02	0.85
			2	Cu	0.8	62	0.0010	0.80	0.06	1	1.6	13.0	1.96	0.0010	1.34	0.07	2.57
			3	Cu	No cap	112	0.0010	0.14	0.02	0	0.0	0.0	1.32	0.0010	0.14	0.02	1.32
			4	Cu	No cap	214	0.0010	0.28	0.04	0	0.0	0.0	1.37	0.0010	0.28	0.04	1.37
			5	Cu	0.4	285	0.0010	0.40	0.05	2	0.7	1.1	1.12	0.0010	0.49	0.05	1.18
			6	Cu	0.5	367	0.0030	0.50	0.04	2	0.5	1.0	1.49	0.0030	0.67	0.04	1.56
			7	Cu	0.5	684	0.0010	0.50	0.03	1	0.1	1.5	1.48	0.0010	0.75	0.03	1.65
			8	Cu	1.0	453	0.0030	1.00	0.04	2	0.4	0.9	1.82	0.0030	1.42	0.04	1.96
			9	Cu	1.0	488	0.0030	1.00	0.05	6	1.2	4.1	2.29	0.0030	1.62	0.05	2.55
			10	Cu	1.5	313	0.0030	2.51	0.13	0	0.0	0.0	2.38	0.0030	2.51	0.13	2.38
			12	Cu	No cap	12	0.0030	0.06	0.02	0	0.0	0.0	0.85	0.0030	0.06	0.02	0.85

							Capped							Uncapped			
Deposit	Area	Domain	Wireframe	Metal	Cap (g/t)	# of Samples	Min	Max	Mean	# Capped	% Capped	% Metal Lost	CV	Min	Max	Mean	CV
		E-W	1	Cu	No cap	293	0.0010	0.47	0.07	0	0.0	0.0	1.44	0.0010	0.47	0.07	1.44
			2	Cu	No cap	51	0.0030	0.19	0.05	0	0.0	0.0	1.00	0.0030	0.19	0.05	1.00
			3	Cu	No cap	241	0.0010	0.46	0.05	0	0.0	0.0	1.19	0.0010	0.46	0.05	1.19
			4	Cu	No cap	25	0.0020	0.07	0.03	0	0.0	0.0	0.82	0.0020	0.07	0.03	0.82
			5	Cu	0.8	78	0.0010	0.80	0.07	1	1.3	2.7	1.55	0.0010	0.99	0.07	1.72
			6	Cu	1.0	520	0.0010	1.00	0.06	1	0.2	1.3	1.33	0.0010	1.34	0.06	1.48
			7	Cu	No cap	542	0.0030	0.65	0.06	0	0.0	0.0	1.30	0.0010	1.00	0.06	1.30
			8	Cu	1.2	312	0.0010	1.20	0.09	3	1.0	1.9	1.87	0.0010	1.62	0.10	1.97
			9	Cu	1.2	542	0.0010	1.20	0.07	1	1.7	7.3	2.58	0.0010	2.50	0.07	2.95
			10	Cu	0.8	347	0.0010	0.80	0.02	3	0.9	26.0	3.59	0.0010	2.10	0.03	5.30
			11	Cu	0.2	127	0.0010	0.20	0.01	1	0.8	6.6	1.95	0.0010	0.42	0.01	2.64
			12	Cu	No cap	87	0.0010	0.23	0.01	0	0.0	0.0	1.84	0.0010	0.23	0.01	1.84
			13	Cu	No cap	161	0.0010	0.26	0.01	0	0.0	0.0	2.82	0.0010	0.26	0.01	2.82
			14	Cu	1.2	213	0.0010	1.20	0.03	2	0.9	27.0	3.92	0.0010	2.96	0.41	5.77
			15	Cu	No cap	107	0.0010	0.04	0.00	0	0.0	0.0	1.52	0.0010	0.04	0.00	1.52
			16	Cu	No cap	67	0.0010	0.07	0.01	0	0.0	0.0	1.56	0.0010	0.07	0.01	1.56

Table 14-16: Toroparu SE, SEHW, and SEFW Area Copper Outlier Analysis and Capping

							Capped							Uncapped			
Deposit	Area	Domain	Wireframe	Metal	Cap (g/t)	# of Samples	Min	Max	Mean	# Capped	% Capped	% Metal Lost	CV	Min	Max	Mean	CV
Toroparu	SE/SEHW/SEFW	Background	1	Cu	1.0	2,850	0.0030	0.12	0.01	0	0.0	0.0	1.72	0.0001	1.06	0.03	1.72
	SE	NW-SE	2	Cu	No cap	5	0.0140	0.78	0.36	0	0.0	0.0	0.99	0.0140	0.78	0.36	0.99
			3	Cu	No cap	9	0.0200	1.81	0.45	0	0.0	0.0	1.49	0.0200	1.81	0.45	1.49
			4	Cu	No cap	3	0.1070	0.71	0.33	0	0.0	0.0	1.01	0.1070	0.71	0.33	1.01
			5	Cu	No cap	2	0.0030	0.08	0.37	0	0.0	0.0	1.52	0.0030	0.08	0.37	1.52
			6	Cu	No cap	4	0.0100	0.19	0.10	0	0.0	0.0	0.76	0.0100	0.19	0.10	0.76
			7	Cu	No cap	4	0.0640	0.06	0.06	0	0.0	0.0	0.00	0.0640	0.06	0.06	0.00
			8	Cu	No cap	3	0.0050	0.01	0.01	0	0.0	0.0	0.18	0.0050	0.01	0.01	0.18
			9	Cu	No cap	4	0.0110	0.83	0.06	0	0.0	0.0	0.55	0.0110	0.83	0.06	0.55
			10	Cu	No cap	3	0.1060	0.22	0.17	0	0.0	0.0	0.34	0.1060	0.22	0.17	0.34
			11	Cu	No cap	4	0.1310	0.33	0.24	0	0.0	0.0	0.35	0.1310	0.33	0.24	0.35
			13	Cu	No cap	7	0.0220	0.14	0.06	0	0.0	0.0	0.72	0.0220	0.14	0.06	0.72
			14	Cu	No cap	3	0.0280	0.09	0.06	0	0.0	0.0	0.52	0.0280	0.09	0.06	0.52
			15	Cu	No cap	4	0.0190	0.25	0.09	0	0.0	0.0	1.25	0.0190	0.25	0.09	1.25
			16	Cu	No cap	4	0.0230	1.39	0.39	0	0.0	0.0	1.68	0.0230	1.39	0.39	1.68
			17	Cu	No cap	2	0.0810	0.17	0.13	0	0.0	0.0	0.51	0.0810	0.17	0.13	0.51
			18	Cu	No cap	2	0.0010	0.00	0.00	0	0.0	0.0	0.11	0.0010	0.00	0.00	0.11
			19	Cu	No cap	3	0.5110	1.11	0.71	0	0.0	0.0	0.48	0.5110	1.11	0.71	0.48
			20	Cu	No cap	2	0.2090	0.21	0.21	0	0.0	0.0	0.01	0.2090	0.21	0.21	0.01
			21	Cu	No cap	7	0.0150	0.05	0.03	0	0.0	0.0	0.57	0.0150	0.05	0.03	0.57
			22	Cu	No cap	2	0.0690	0.09	0.08	0	0.0	0.0	0.16	0.0690	0.09	0.08	0.16
			23	Cu	No cap	2	0.8980	1.48	1.18	0	0.0	0.0	0.34	0.8980	1.48	1.18	0.34
			24	Cu	No cap	3	0.0550	0.83	0.41	0	0.0	0.0	0.97	0.0550	0.83	0.41	0.97
			25	Cu	No cap	5	0.0350	0.29	0.18	0	0.0	0.0	0.56	0.0350	0.29	0.18	0.56
			26	Cu	No cap	3	0.0040	0.06	0.03	0	0.0	0.0	1.07	0.0040	0.06	0.03	1.07
		E-W	1	Cu	No cap	97	0.0010	0.46	0.05	0	0.0	0.0	1.65	0.0010	0.46	0.05	1.65
			2	Cu	No cap	243	0.0010	1.00	0.12	3	1.2	4.7	1.57	0.0010	1.82	0.13	1.76
			3	Cu	No cap	38	0.0010	0.63	0.20	0	0.0	0.0	0.80	0.0010	0.63	0.20	0.80
			4	Cu	No cap	137	0.0020	2.26	0.09	1	0.7	6.7	2.30	0.0020	2.26	0.09	2.69
			5	Cu	No cap	132	0.0010	0.35	0.05	0	0.0	0.0	1.43	0.0010	0.35	0.05	1.43
	SEHW	NW-SE	2	Cu	No cap	81	0.0040	0.14	0.04	0	0.0	0.0	0.82	0.0040	0.14	0.04	0.82
			3	Cu	No cap	7	0.0080	0.30	0.09	0	0.0	0.0	1.05	0.0080	0.30	0.09	1.05
			4	Cu	No cap	157	0.0020	0.32	0.06	0	0.0	0.0	1.09	0.0020	0.32	0.06	1.09

							Capped							Uncapped			
Deposit	Area	Domain	Wireframe	Metal	Cap (g/t)	# of Samples	Min	Max	Mean	# Capped	% Capped	% Metal Lost	CV	Min	Max	Mean	CV
			5	Cu	No cap	15	0.0050	0.08	0.02	0	0.0	0.0	0.78	0.0050	0.08	0.02	0.78
			6	Cu	No cap	8	0.0100	0.08	0.04	0	0.0	0.0	0.58	0.0100	0.08	0.04	0.58
			7	Cu	No cap	7	0.0020	0.22	0.06	0	0.0	0.0	1.36	0.0020	0.22	0.06	1.36
			8	Cu	No cap	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
			9	Cu	No cap	3	0.0060	0.24	0.11	0	0.0	0.0	1.05	0.0060	0.24	0.11	1.05
			10	Cu	No cap	4	0.1010	0.51	0.30	0	0.0	0.0	0.65	0.1010	0.51	0.30	0.65
			11	Cu	No cap	81	0.0040	0.14	0.04	0	0.0	0.0	0.82	0.0040	0.14	0.04	0.82
			12	Cu	No cap	7	0.0080	0.30	0.09	0	0.0	0.0	1.05	0.0080	0.30	0.09	1.05
			13	Cu	No cap	157	0.0020	0.32	0.06	0	0.0	0.0	1.09	0.0020	0.32	0.06	1.09
	SEFW	NW-SE	2	Cu	No cap	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
			3	Cu	No cap	35	0.0020	0.04	0.02	0	0.0	0.0	0.76	0.0020	0.04	0.02	0.00
			4	Cu	No cap	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
			6	Cu	0.3	254	0.0020	0.30	0.55	2	0.8	0.2	0.98	0.0020	0.33	0.55	0.99
			7	Cu	No cap	375	0.0010	0.44	0.03	0	0.0	0.0	1.35	0.0010	0.44	0.03	1.35
			8	Cu	No cap	4	0.0040	0.01	0.01	0	0.0	0.0	0.63	0.0040	0.01	0.01	0.63
			9	Cu	No cap	12	0.0060	0.03	0.01	0	0.0	0.0	0.55	0.0060	0.03	0.01	0.55
			10	Cu	No cap	2	0.0180	0.03	0.02	0	0.0	0.0	0.23	0.0180	0.03	0.02	0.23
			11	Cu	No cap	3	0.0100	0.05	0.02	0	0.0	0.0	0.88	0.0100	0.05	0.02	0.88
			12	Cu	No cap	2	0.0210	0.04	0.02	0	0.0	0.0	0.41	0.0210	0.04	0.02	0.41
			13	Cu	No cap	7	0.0010	0.00	0.00	0	0.0	0.0	0.42	0.0010	0.00	0.00	0.42

Table 14-17: Toroparu Main and NW Area Silver Outlier Analysis and Capping

							Capped							Uncapped			
Deposit	Area	Domain	Wireframe	Metal	Cap (g/t)	# of Samples	Min	Max	Mean	# Capped	% Capped	% Metal Lost	CV	Min	Max	Mean	CV
Toroparu	Main/NW	Background	1	Ag	60.0	21,415	0.0000	60.00	0.76	1	0.0	0.3	1.86	0.0000	101.00	0.76	2.02
	Main	NW-SE	1	Ag	No cap	106	0.0030	7.00	0.82	0	0.0	0.0	1.53	0.0030	7.00	0.82	1.53
			2	Ag	No cap	38	0.0600	12.80	2.16	0	0.0	0.0	1.26	0.0600	12.80	2.16	1.26
			3	Ag	No cap	431	0.0200	14.83	2.05	0	0.0	0.0	1.19	0.0200	14.83	2.05	1.19
			4	Ag	60.0	556	0.0100	60.00	1.65	1	0.2	0.7	1.66	0.0100	75.49	1.66	1.81
			5	Ag	No cap	1,138	0.0100	21.00	1.62	0	0.0	0.0	1.19	0.0100	21.00	1.62	1.19
			6	Ag	60.0	2,123	0.0200	60.00	2.78	3	0.1	0.7	1.39	0.0200	86.11	2.80	1.50
			7	Ag	No cap	726	0.0100	13.93	1.23	0	0.0	0.0	1.26	0.0100	13.93	1.23	1.26
			8	Ag	45.0	392	0.0030	45.00	1.29	2	0.5	0.7	2.09	0.0030	49.75	1.30	2.20
			9	Ag	No cap	53	0.0200	6.73	1.55	0	0.0	0.0	0.94	0.0200	6.73	1.55	0.94

							Capped								Uncapped			
Deposit	Area	Domain	Wireframe	Metal	Cap (g/t)	# of Samples	Min	Max	Mean	# Capped	% Capped	% Metal Lost	CV	Min	Max	Mean	CV
		E-W	1	Ag	No cap	206	0.0100	14.36	1.43	0	0.0	0.0	1.58	0.0100	14.36	1.43	1.58
			2	Ag	No cap	174	0.0200	4.00	0.70	0	0.0	0.0	1.16	0.0200	4.00	0.70	1.16
			3	Ag	No cap	274	0.0200	8.48	1.88	0	0.0	0.0	0.92	0.0200	8.48	1.88	0.92
			4	Ag	No cap	460	0.0800	27.00	2.98	0	0.0	0.0	1.07	0.0800	27.00	2.98	1.07
			5	Ag	60.0	1,457	0.0300	60.00	3.06	2	0.1	1.0	1.37	0.0300	86.11	3.09	1.52
			6	Ag	No cap	21	0.0500	21.00	2.26	0	0.0	0.0	1.08	0.0500	21.00	2.26	1.08
			7	Ag	No cap	584	0.0300	14.83	1.09	0	0.0	0.0	1.22	0.0300	14.83	1.09	1.22
			8	Ag	No cap	322	0.0100	22.18	1.33	0	0.0	0.0	1.43	0.0100	22.18	1.33	1.43
			9	Ag	No cap	128	0.0400	10.17	1.08	0	0.0	0.0	1.29	0.0400	10.17	1.08	1.29
			10	Ag	No cap	233	0.0100	13.10	0.76	0	0.0	0.0	1.47	0.0100	13.10	0.76	1.47
			11	Ag	30.0	125	0.0200	30.00	1.99	2	1.6	6.4	2.12	0.0200	49.75	2.13	2.50
			12	Ag	No cap	65	0.0300	3.79	0.74	0	0.0	0.0	1.04	0.0300	3.79	0.74	1.04
	NW	NW-SE	1	Ag	No cap	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
			2	Ag	No cap	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
			3	Ag	No cap	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
			4	Ag	No cap	7	0.0600	0.23	0.14	0	0.0	0.0	0.52	0.0600	0.23	0.14	0.52
			5	Ag	No cap	3	0.6400	11.67	5.46	0	0.0	0.0	1.03	0.6400	11.67	5.46	1.03
			6	Ag	No cap	11	0.0400	0.54	0.23	0	0.0	0.0	0.98	0.0400	0.54	0.23	0.98
			7	Ag	No cap	57	0.0100	5.60	0.50	0	0.0	0.0	1.91	0.0100	5.60	0.50	1.91
			8	Ag	15.0	74	0.0200	15.00	0.95	1	1.4	8.1	1.91	0.0200	21.90	1.04	2.42
			9	Ag	No cap	122	0.0100	4.20	0.60	0	0.0	0.0	1.03	0.0100	4.20	0.60	1.03
			10	Ag	No cap	104	0.0100	10.90	0.78	0	0.0	0.0	1.90	0.0100	10.90	0.78	1.90
			12	Ag	No cap	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
		E-W	1	Ag	No cap	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
			2	Ag	No cap	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
			3	Ag	No cap	8	0.2400	2.52	0.63	0	0.0	0.0	1.23	0.2400	2.52	0.63	1.23
			4	Ag	No cap	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
			5	Ag	No cap	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
			6	Ag	15.0	16	0.0600	15.00	1.60	1	6.3	12.0	2.57	0.0600	17.55	1.84	2.67
			7	Ag	No cap	88	0.0200	6.54	1.12	0	0.0	0.0	1.10	0.0200	6.54	1.12	1.10
			8	Ag	No cap	71	0.0400	6.49	0.79	0	0.0	0.0	1.33	0.0400	6.49	0.79	1.33
			9	Ag	No cap	47	0.1000	10.90	0.83	0	0.0	0.0	2.25	0.1000	10.90	0.83	2.25
			10	Ag	12.0	41	0.0200	12.00	0.93	1	2.4	9.4	2.58	0.0200	14.52	1.03	2.79
			11	Ag	No cap	14	0.0400	4.20	0.41	0	0.0	0.0	2.17	0.0400	4.20	0.41	2.17

							Capped							Uncapped			
Deposit	Area	Domain	Wireframe	Metal	Cap (g/t)	# of Samples	Min	Max	Mean	# Capped	% Capped	% Metal Lost	CV	Min	Max	Mean	CV
			12	Ag	No cap	16	0.0600	7.13	1.64	0	0.0	0.0	1.49	0.0600	7.13	1.64	1.49
			13	Ag	No cap	14	0.0500	1.07	0.14	0	0.0	0.0	1.62	0.0500	1.07	0.14	1.62
			14	Ag	No cap	38	0.0300	7.81	0.51	0	0.0	0.0	2.62	0.0300	7.81	0.51	2.62
			15	Ag	No cap	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
			16	Ag	No cap	3	0.6900	3.77	1.79	0	0.0	0.0	1.01	0.6900	3.77	1.79	1.01

Table 14-18: Toroparu SE, SEHW, and SEFW Area Silver Outlier Analysis and Capping

							Capped							Uncapped			
Deposit	Area	Domain	Wireframe	Metal	Cap (g/t)	# of Samples	Min	Max	Mean	# Capped	% Capped	% Metal Lost	CV	Min	Max	Mean	CV
Toroparu	SE/SEHW/SEFW	Background	1	Ag	No cap	0	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
	SE	NW-SE	2	Ag	No cap	0	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
			3	Ag	No cap	4	1.4200	1.42	1.42	0	0.0	0.0	0.00	1.4200	1.42	1.42	0.00
			4 to 9	Ag	No cap	0	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
			10	Ag	No cap	3	0.5500	0.97	0.78	0	0.0	0.0	0.28	0.5500	0.97	0.78	0.28
			11	Ag	No cap	4	1.0600	2.72	2.06	0	0.0	0.0	0.34	1.0600	2.72	2.06	0.34
			13-26	Ag	No cap	0	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
		E-W	1	Ag	No cap	0	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
			2	Ag	No cap	19	0.0300	9.17	3.12	0	0.0	0.0	0.84	0.0300	9.17	3.12	0.84
			3	Ag	No cap	35	0.1400	6.36	1.91	0	0.0	0.0	0.83	0.1400	6.36	1.91	0.83
			4	Ag	No cap	0	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
			5	Ag	No cap	55	0.0300	1.63	0.23	0	0.0	0.0	1.37	0.0300	1.63	0.23	1.37
	SEHW	NW-SE	2 to 13	Ag	No cap	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
	SEFW	NW-SE	2 to 13	Ag	No cap	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a

Table 14-19: Sona Hill Deposit Silver Outlier Analysis and Capping

						Capped							Uncapped			
Deposit	Domain	Wireframe	Metal	Cap (g/t)	# of Samples	Min	Max	Mean	# Capped	% Capped	% Metal Lost	CV	Min	Max	Mean	CV
Sona Hill	Background	1	Ag	3.5	15,321	0.0030	2.00	0.09	17	0.1	2.0	2.30	0.0030	7.02	0.09	2.55
	NW-SE	1	Ag	10.0	94	0.0400	9.75	1.19	1	1.1	1.2	1.73	0.0400	10.46	1.20	1.76
		2	Ag	7.0	146	0.0100	9.00	0.96	2	1.4	1.2	1.26	0.0100	9.00	0.97	1.31
		3	Ag	10.0	171	0.0100	10.00	1.27	2	1.2	3.5	1.56	0.0100	15.82	1.31	1.70
		4	Ag	15.0	79	0.0100	15.00	1.22	4	5.1	24.0	1.95	0.0100	48.30	1.61	3.33
		5	Ag	5.0	59	0.0800	5.00	0.89	1	1.7	35.0	1.22	0.0800	52.21	1.36	3.87

						Capped							Uncapped			
Deposit	Domain	Wireframe	Metal	Cap (g/t)	# of Samples	Min	Max	Mean	# Capped	% Capped	% Metal Lost	CV	Min	Max	Mean	CV
		6	Ag	No cap	0	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
		7	Ag	No cap	0	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
		8	Ag	No cap	16	0.0500	1.45	0.49	0	0.0	0.0	0.93	0.0500	1.45	0.49	0.93
		9	Ag	No cap	1	0.1600	0.16	0.16	0	0.0	0.0	0.00	0.1600	0.16	0.16	0.00
		10	Ag	5.0	32	0.0200	6.00	1.07	2	6.3	14.0	1.60	0.0200	11.29	1.25	1.91
		11	Ag	2.5	43	0.0500	2.50	0.72	4	9.3	29.0	1.01	0.0500	7.75	1.02	1.71
		12	Ag	No cap	25	0.0100	4.65	0.66	0	0.0	0.0	1.69	0.0100	4.65	0.66	1.69
		13	Ag	No cap	7	0.0900	3.35	1.70	0	0.0	0.0	0.79	0.0900	3.35	1.70	0.79
		14	Ag	No cap	13	0.0800	6.41	1.30	0	0.0	0.0	1.46	0.0800	6.41	1.30	1.46
	E-W	1	Ag	No cap	8	0.1300	2.65	0.67	0	0.0	0.0	1.84	0.1300	2.65	0.67	1.84
		2	Ag	5.0	29	0.0900	5.00	0.75	2	6.9	15.0	1.66	0.0030	7.83	0.88	1.99
		3	Ag	No cap	12	0.0400	3.69	0.90	0	0.0	0.0	1.52	0.0400	3.69	0.90	1.52
		4	Ag	No cap	3	0.1200	3.43	1.66	0	0.0	0.0	1.40	0.1200	3.43	1.66	1.40
		5	Ag	No cap	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
		6	Ag	No cap	8	0.0800	2.67	0.61	0	0.0	0.0	1.59	0.0800	2.67	0.61	1.59
		7	Ag	15.0	4	0.4000	15.00	2.27	1	25.0	62.0	2.20	0.4000	52.21	6.00	3.03
		8	Ag	No cap	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
		9	Ag	15.0	71	0.2500	15.00	1.14	1	1.4	23.0	1.73	0.2500	48.30	1.49	3.39
		10	Ag	No cap	25	0.0400	4.30	0.82	0	0.0	0.0	1.43	0.0400	4.30	0.82	1.43
		11	Ag	5.0	62	0.2100	4.40	0.95	0	0.0	0.0	0.96	0.2100	4.40	0.95	0.96

14.4.2 Compositing

Compositing of assays is a technique used to give each sample a relatively equal length to reduce the potential for bias due to uneven sample lengths; it prevents the potential loss of sample data and reduces the potential for grade bias due to the possible creation of short and potentially high-grade composites that are generally formed along the zone contacts when using a fixed length.

The raw sample data was found to have a moderately consistent range of sample lengths. Samples captured within all Domains were composited to 1.0 m regular intervals based on the observed modal distribution of sample lengths, which supports a 5.0 m x 5.0 m x 5.0 m block model (Northing x Easting x Elevation) with two sub blocking levels (a minimum size of Northing = 1.25 m x Easting = 1.25 m x Variable Elevation). An option to use a slightly variable composite length was chosen to allow for backstitching shorter composites located along the edges of the composited interval. All composite samples were generated within each background low-grade, northwest-southeast, and east-west wireframe. There are no overlaps along boundaries. The composite samples were statistically validated to ensure no material loss of data or change to each sample population's mean grade. Table 14-20 through Table 14-24 summarize the composite counts for all wireframes.

Table 14-20: Toroparu and Sona Hill Deposit Background Low-Grade Composite Counts for Gold, Copper, and Silver by Area

Toroparu Deposit				
Background Low-Grade Domain				
Area	Wireframe	Composite Counts		
		Au	Cu	Ag
Main/NW	1	150,725	136,649	27,916
SE	1	8,200	8,011	0
Sona Hill Deposit				
Background Domain				
Area	**Wireframe**	**Composite Counts**		
		Au	**Cu**	**Ag**
Sona Hill	1	17,867	3,516	3,516

Table 14-21: Toroparu Deposit, Main, and NW Area Composite Counts by Domain Wireframe for Gold, Copper, and Silver

Toroparu Deposit									
NW-SE Domain					E-W Domain				
Area	Wireframe	Composite Counts			Area	Wireframe	Composite Counts		
		Au	Cu	Ag			Au	Cu	Ag
Main	1	946	944	139	Main	1	722	717	171
	2	246	235	52		2	656	646	162
	3	1,757	1,750	586		3	892	885	342
	4	1,504	1,501	723		4	1,340	1,338	615
	5	2,953	2,908	1,409		5	2,666	2,618	1,845
	6	5,056	5,042	2,573		6	675	664	239
	7	2,772	2,755	896		7	1,358	1,346	744
	8	2,342	2,276	461		8	761	761	400
	9	560	531	75		9	492	486	190
						10	849	844	244
						11	743	743	143
						12	646	646	92
NW	1	18	18	0	NW	1	405	405	0
	2	94	94	0		2	73	73	0
	3	159	154	0		3	326	317	12
	4	269	269	5		4	33	33	0
	5	352	352	4		5	108	108	0
	6	455	455	11		6	666	658	21
	7	897	876	59		7	675	646	82
	8	636	592	65		8	401	377	74
	9	634	599	113		9	711	681	47
	11	44	39	6		10	502	467	40
						11	169	169	11
						12	102	102	12
						13	248	234	20
						14	287	277	37
						15	162	147	0
						16	99	99	3

Table 14-22: Toroparu Deposit, SE Area Composite Counts by Domain Wireframe for Gold, Copper, and Silver

Toroparu Deposit									
NW-SE Domain					E-W Domain				
Area	Wireframe	Composite Counts			Area	Wireframe	Composite Counts		
		Au	Cu	Ag			Au	Cu	Ag
SE	2	8	8	0	SE	1	139	139	0
	3	9	9	1		2	342	342	30
	4	5	5	0		3	53	53	48
	5	4	4	0		4	189	189	0
	6	6	6	0		5	170	170	71
	7	2	2	0					
	8	5	5	0					
	9	7	7	0					
	10	5	5	5					
	11	6	6	6					
	13	11	11	0					
	14	5	5	0					
	15	6	6	0					
	16	6	6	0					
	17	3	3	0					
	18	3	3	0					
	19	5	5	0					
	20	3	3	0					
	21	6	6	6					
	22	11	11	0					
	23	5	5	0					
	24	6	6	0					
	25	6	6	0					
	26	3	3	0					

Table 14-23: Toroparu Deposit, SEHW, and SEFW Area Composite Counts by Domain Wireframe for Gold, Copper, and Silver

<table>
<tr><th colspan="10">Toroparu Deposit</th></tr>
<tr><th colspan="5">NW-SE Domain</th><th colspan="5">E-W Domain</th></tr>
<tr><th rowspan="2">Area</th><th rowspan="2">Wireframe</th><th colspan="3">Composite Counts</th><th rowspan="2">Area</th><th rowspan="2">Wireframe</th><th colspan="3">Composite Counts</th></tr>
<tr><th>Au</th><th>Cu</th><th>Ag</th><th>Au</th><th>Cu</th><th>Ag</th></tr>
<tr><td rowspan="9">SEHW</td><td>2</td><td>125</td><td>112</td><td>0</td><td colspan="5" rowspan="9">No East-West Domain Wireframes exist in the SEHW Domain</td></tr>
<tr><td>3</td><td>9</td><td>9</td><td>0</td></tr>
<tr><td>4</td><td>193</td><td>193</td><td>0</td></tr>
<tr><td>5</td><td>23</td><td>23</td><td>0</td></tr>
<tr><td>6</td><td>7</td><td>7</td><td>0</td></tr>
<tr><td>7</td><td>9</td><td>9</td><td>0</td></tr>
<tr><td>8</td><td>2</td><td>0</td><td>0</td></tr>
<tr><td>9</td><td>5</td><td>5</td><td>0</td></tr>
<tr><td>10</td><td>5</td><td>5</td><td>0</td></tr>
<tr><td rowspan="11">SEFW</td><td>2</td><td>9</td><td>9</td><td>0</td><td colspan="5" rowspan="11">No East-West Domain Wireframes exist in the SEFW Domain</td></tr>
<tr><td>3</td><td>49</td><td>49</td><td>0</td></tr>
<tr><td>4</td><td>6</td><td>6</td><td>0</td></tr>
<tr><td>6</td><td>359</td><td>359</td><td>0</td></tr>
<tr><td>7</td><td>501</td><td>501</td><td>0</td></tr>
<tr><td>8</td><td>6</td><td>6</td><td>0</td></tr>
<tr><td>9</td><td>12</td><td>12</td><td>0</td></tr>
<tr><td>10</td><td>3</td><td>3</td><td>0</td></tr>
<tr><td>11</td><td>5</td><td>5</td><td>0</td></tr>
<tr><td>12</td><td>3</td><td>3</td><td>0</td></tr>
<tr><td>13</td><td>6</td><td>6</td><td>0</td></tr>
</table>

Table 14-24: Sona Hill Deposit Composite Counts by Domain Wireframe for Gold, Copper, and Silver

Sona Hill Deposit									
NW-SE Domain					E-W Domain				
Area	Wireframe	Composite Counts			Area	Wireframe	Composite Counts		
		Au	Cu	Ag			Au	Cu	Ag
	1	508	92	92		1	159	5	5
	2	793	155	155		2	201	43	43
	3	821	143	143		3	239	14	14
	4	335	74	74		4	33	5	5
	5	289	53	53		5	26	0	0
	6	13	0	0		6	103	8	8
	7	21	0	0		7	84	7	7
	8	92	13	13		8	118	2	2
	9	27	2	2		9	191	71	71
	10	297	44	44		10	94	31	31
	11	163	51	51		11	107	60	60
	12	123	27	27					
	13	92	12	12					
	14	77	16	16					

14.4.3 Specific Gravity

A total of 6,169 SG measurements exist for the Toroparu Deposit, and 723 SG measurements exist for the Sona Hill Deposit, provided from laboratory measurements. Measurements were taken from DDH samples using the weight in air versus the weight in water method (Archimedes) by applying the following formula:

$$Specific\ Gravity = \frac{Weight\ in\ Air}{(Weight\ in\ Air - Weight\ in\ Water)}$$

Nordmin determined that SG measurements' required amount and distribution did not exist for direct estimation of the entire block model for both the Toroparu Deposit and Sona Hill Deposits.

- Toroparu Deposit: SG weighted averages were calculated for saprolite. Non saprolite material was subdivided into the Domains within lithological groups. SG weighted average values were calculated and applied as described in Table 14-25.
- Sona Hill Deposit: SG weighted averages were calculated for saprolite. Non saprolite material was subdivided into lithological groups. SG weighted average values were calculated and applied as described in Table 14-25.

Table 14-25: SG Assignment

Toroparu Deposit				Sona Hill Deposit		
Saprolite/Non Saprolite	**Lithological Group**	**Domain**	**SG**	**Saprolite/Non Saprolite**	**Lithological Group**	**SG**
Saprolite	n/a	n/a	1.860	Saprolite	n/a	1.742
Fresh	Volcanics	Background	2.720	Fresh	CATHYDR	2.822
		NW-SE	2.700		Intrusive	2.807
		E-W	2.730		Volcanic	2.748
	Intrusive	Background	2.710		Other	2.758
		NW-SE	2.730			
		E-W	2.780			

14.5 Block Model Resource Estimation

14.5.1 Block Model Strategy and Analysis

A series of upfront test modelling was completed to define an estimation methodology to meet the following criteria:

- representative of the deposit geology, structural models, and geological controls on mineralization,
- accounts for the variability of grade, orientation, and continuity of mineralization,
- controls the smoothing (grade spreading) of grades and the influence of outliers,
- accounts for most of the mineralization,
- is robust and repeatable within the deposits and Domains, and

- supports multiple NW-SE, E-W, and Low-Grade Background Domains.

Multiple test scenarios were evaluated to determine the optimum processes and parameters to achieve the stated criteria. Each scenario was based on nearest neighbour (NN), inverse distance squared (ID2), inverse distance cubed (ID3), and ordinary kriging (OK) interpolation methods.

All test scenarios were evaluated based on global statistical comparisons, visual comparisons of composite samples versus block grades, and overall smoothing assessment. Based on the testing results, it was determined that all scenarios, including the draft and final resource estimation methodology, would constrain the mineralization by using hard wireframe boundaries to control the spread of high-grade and background low-grade mineralization. Therefore, OK was selected as the most representative interpolation method.

14.5.2 Block Model Definition

Block model shape and size are typically a function of the geometry of the deposit, the density of sample data, drill hole spacing, and the selected mining unit. Block models were defined with parent blocks at 5.0 m x 5.0 m x 5.0 m (N-S x E-W x Elevation). Block model parameters are defined in Table 14-26.

All wireframe volumes were filled with blocks from the prototype (which used the parameters in Table 14-26). Block volumes were compared to the wireframe volumes to confirm there were no significant differences. Block volumes for all wireframes were found to be within reasonable tolerance limits. Sub blocking was implemented to maintain the geological interpretation and accommodate the background, NW-SE, and E-W Domains (wireframes), the SG, and the category application. Sub blocking has been allowed to the following minimums:

- 5.0 m x 5.0 m x 5.0 m blocks are sub blocked two-fold to 1.25 m x 1.25 m in the N-S and E-W directions, with a variable elevation calculated based on the other sizes.

Table 14-26: Block Model Definition

Toroparu Deposit						
Item	**Block Origin**	**Block Maximum**	**Block Extent (m)**	**Block Dimension (m)**	**Number of Blocks**	**Minimum Sub Block (m)**
Easting	821,500	829,500	8,000	5.0	1,600	1.25
Northing	711,400	717,500	6,100	5.0	1,220	1.25
Elevation	-1,000	300	1,300	5.0	260	Variable
Sona Hill Deposit						
Item	**Block Origin**	**Block Maximum**	**Block Extent (m)**	**Block Dimension (m)**	**Number of Blocks**	**Minimum Sub Block (m)**
Easting	829,950	830,950	1,000	5.0	200	1.25
Northing	710,200	711,800	1,600	5.0	320	1.25
Elevation	-350	250	600	5.0	120	Variable

Block models were not rotated but were clipped to topography. The Mineral Resource Estimate was conducted using Datamine Studio RMTM version 1.8.37.0 within the PSAD56 Zone 20 North datum.

Two block models were independently estimated, one each for the Toroparu and Sona Hill Deposits.

14.5.3 Interpolation Method

The Project block models were estimated using NN, ID2, ID3, and OK interpolation methods for global comparisons and validation purposes. The OK method was selected over NN, ID2, and ID3 for the Mineral Resource Estimate as the method best controlling estimation and smoothing of grades and was the most representative of all deposits in the Project.

14.5.4 Search Strategy

Zonal controls were used to constrain the grade estimates to each wireframe. These controls prevented the samples from individual wireframes from influencing the block grades of others, acting as a "hard boundary" between the wireframes.

The search orientation strategy determined to be most representative of the mineralization at the deposits was to use a combination of a search ellipsoid for each grade within each domain and allow dynamic anisotropy in the estimation process. Dynamic anisotropy was applied during the estimation process to the NW-SE and E-W Domains of both the Toroparu and Sona Hill Deposits, which adjusts the search ellipsoid on a block by block basis controlled by the orientation of all mineralized wireframes. It is Nordmin's opinion that dynamic anisotropy allows for a much more accurate estimation of grade and mineralization due to the nature of the higher-grade and lower grade Domains.

Estimation passes were defined with carefully-selected search distances. The first pass is correlated to a Measured categorization. The second pass is correlated to an Indicated categorization, and the third pass is correlated to an Inferred categorization. Overall search parameters can be found in Table 14-27.

Table 14-27: Search Parameters

				Ellipsoid Rotation Angles				Ranges, Search Pass 1 (m)			Ranges, Search Pass 2 (m)			Ranges, Search Pass 3 (m)			Composites, Pass 1		Composites, Pass 2		Composites, Pass 3	
Area	Domain	Metal	Dynamic Anisotropy	1	2	3	Axes	1	2	3	1	2	3	1	2	3	Min	Max	Min	Max	Min	Max
Toroparu Deposit																						
Main/NW	Background	All	N	30	-88	Z-X-Z	0	30	30	15	60	60	30	180	180	90	3	8	3	8	2	8
SE	Background	All	N	20	-80	Z-X-Z	0	30	30	15	60	60	30	180	180	90	3	8	3	8	2	8
Main	LG	Au	Y	30	-88	Z-X-Z	25	30	30	15	60	60	30	180	180	90	3	8	3	8	2	8
Main	HG	Au	Y	0	-90	Z-X-Z	15	30	30	15	60	60	30	180	180	90	3	8	3	8	2	8
Main	LG	Ag	Y	30	-88	Z-X-Z	5	30	15	30	60	30	60	180	90	180	3	8	3	8	2	8
Main	HG	Ag	Y	0	-90	Z-X-Z	15	30	15	30	60	30	60	180	90	180	3	8	3	8	2	8
Main	LG	Cu	Y	-15	-88	Z-X-Z	5	15	30	30	30	60	60	90	180	180	3	8	3	8	2	8
Main	HG	Cu	Y	0	-90	Z-X-Z	15	15	30	30	30	60	60	90	180	180	3	8	3	8	2	8
NW	LG	Au	Y	30	-90	Z-X-Z	15	30	30	15	60	60	30	180	180	90	3	8	3	8	2	8
NW	HG	Au	Y	7	-88	Z-X-Z	0	30	30	15	60	60	30	180	180	90	3	8	3	8	2	8
NW	LG	Ag	Y	30	-90	Z-X-Z	15	30	15	30	60	30	60	180	90	180	3	8	3	8	2	8
NW	HG	Ag	Y	7	-88	Z-X-Z	0	30	15	30	60	30	60	180	90	180	3	8	3	8	2	8
NW	LG	Cu	Y	30	-90	Z-X-Z	15	15	30	30	30	60	60	90	180	180	3	8	3	8	2	8
NW	HG	Cu	Y	7	-88	Z-X-Z	0	15	30	30	30	60	60	90	180	180	3	8	3	8	2	8
SE, SEHW, SEFW	LG	Au	Y	20	-80	Z-X-Z	15	30	30	15	60	60	30	180	180	90	3	8	3	8	2	8
SE, SEHW, SEFW	HG	Au	Y	-18	83	Z-X-Z	0	30	30	15	60	60	30	180	180	90	3	8	3	8	2	8

				Ellipsoid Rotation Angles				Ranges, Search Pass 1 (m)			Ranges, Search Pass 2 (m)			Ranges, Search Pass 3 (m)			Composites, Pass 1		Composites, Pass 2		Composites, Pass 3	
Area	**Domain**	**Metal**	**Dynamic Anisotropy**	**1**	**2**	**3**	**Axes**	**1**	**2**	**3**	**1**	**2**	**3**	**1**	**2**	**3**	**Min**	**Max**	**Min**	**Max**	**Min**	**Max**
SE, SEHW, SEFW	LG	Ag	Y	20	-80	Z-X-Z	15	30	15	30	60	30	60	180	90	180	3	8	3	8	2	8
SE, SEHW, SEFW	HG	Ag	Y	-18	83	Z-X-Z	0	30	15	30	60	30	60	180	90	180	3	8	3	8	2	8
SE, SEHW, SEFW	LG	Cu	Y	20	-80	Z-X-Z	15	15	30	30	30	60	60	90	180	180	3	8	3	8	2	8
SE, SEHW, SEFW	HG	Cu	Y	-18	83	Z-X-Z	0	15	30	30	30	60	60	90	180	180	3	8	3	8	2	8
Sona Hill Deposit																						
Sona Hill	Background	All	Y	90	-36	Z-X-Z	0	30	20	20	60	40	40	180	120	120	3	8	3	8	2	8
Sona Hill	LG	Au	Y	65	-90	Z-Y-Z	0	30	15	30	60	30	60	180	90	180	3	8	3	8	2	8
Sona Hill	LG	Ag	Y	65	-90	Z-Y-Z	0	30	15	30	60	30	60	180	90	180	3	8	3	8	2	8
Sona Hill	LG	Cu	Y	65	-90	Z-Y-Z	0	30	15	30	60	30	60	180	90	180	3	8	3	8	2	8
Sona Hill	HG	Au	Y	65	-90	Z-Y-Z	0	30	15	30	60	30	60	180	90	180	3	8	3	8	2	8
Sona Hill	HG	Ag	Y	65	-90	Z-Y-Z	0	30	15	30	60	30	60	180	90	180	3	8	3	8	2	8

14.5.5 Assessment of Spatial Grade Continuity

Datamine, X10 Geo, and Sage 2001 were used to determine the geostatistical relationships of each Deposit. Independent variography was performed on composite data for each wireframe within each Deposit. Experimental variograms were calculated from the capped/composited sample gold data to determine the approximate search ellipse dimensions and orientations.

The analyses considered the following:

- Downhole variograms were created and modelled to define the nugget effect.
- Experimental pairwise relative correlogram variograms were calculated to determine directional variograms for the strike and down dip orientations.
- Variograms were modelled using an exponential width practical range.
- Directional variograms were modelled using the nugget defined in the downhole variography and the ranges for strike, perpendicular to strike, and down dip directions.
- Variogram outputs were re-oriented to reflect the orientation of the mineralization.
- Individual variograms were created for each high-grade mineralized belt and low-grade Domain.

Variography parameters used are provided in Table 14-28.

Table 14-28: Variography Parameters

Area	Domain	Metal	Ellipsoid Angles			Nugget	Structure 1 Ranges (m)			C1	Structure 2 Ranges (m)			C2
			1 (X)	2 (Y)	3 (Z)		1	2	3		1	2	3	
Toroparu Deposit														
Main/NW	Background	Au	16	-29	3	0.196	6.5	6.9	45.5	0.538	580	730	205	0.267
Main/NW	Background	Ag	-27	13	-47	0.154	34	10.5	47.4	0.631	776	323	242	0.215
Main/NW	Background	Cu	-46	46	1	0.160	13.8	15.8	34.5	0.541	797	190	370	0.299
Main	NW-SE	Au	-58	-29	-52	0.032	4.1	23.6	4	0.900	600	100	30	0.068
Main	NW-SE	Ag	-58	-41	92	0.321	70.2	54.3	9.8	0.396	696	357	52	0.290
Main	NW-SE	Cu	-45	59	6	0.213	24.2	7.4	71.3	0.468	1,342	262	396	0.319
Main	E-W	Au	-63	-67	3	0.040	3.4	10.8	6.5	0.900	333	451	69	0.061
Main	E-W	Ag	-20	5	40	0.440	77.2	78.5	7.4	0.313	309	148	72	0.247
Main	E-W	Cu	-152	93	27	0.278	8.1	15.1	59.1	0.348	1,100	380	141	0.374
NW	NW-SE	Au	-58	-29	-52	0.032	4.1	23.6	4	0.900	600	100	30	0.068
NW	NW-SE	Ag	-58	-41	92	0.321	70.2	54.3	9.8	0.396	696	357	52	0.290
NW	NW-SE	Cu	-45	59	6	0.213	24.2	7.4	71.3	0.468	1,342	262	396	0.319
NW	E-W	Au	-63	-67	3	0.040	3.4	10.8	6.5	0.900	333	451	69	0.061
NW	E-W	Ag	-20	5	40	0.440	77.2	78.5	7.4	0.313	309	148	72	0.247
NW	E-W	Cu	-152	93	27	0.278	8.1	15.1	59.1	0.348	1,100	380	141	0.374
SE	Background	Au	-64	-42	22	0.110	3.7	31.2	16.3	0.667	384	141	77	0.223
SE	Background	Ag	-27	13	-47	0.154	34	10.5	47.4	0.631	776	323	242	0.215
SE	Background	Cu	-107	21	61	0.014	12	32	7.1	0.738	92	270	66	0.248
SE	NW-SE	Au	-58	-29	-52	0.032	4.1	23.6	4	0.900	600	100	30	0.068
SE	NW-SE	Ag	-58	-41	92	0.321	70.2	54.3	9.8	0.396	696	357	52	0.290
SE	NW-SE	Cu	-45	59	6	0.213	24.2	7.4	71.3	0.468	1,342	262	396	0.319
SE	E-W	Au	-63	-67	3	0.040	3.4	10.8	6.5	0.900	333	451	69	0.061
SE	E-W	Ag	-20	5	40	0.440	77.2	78.5	7.4	0.313	309	148	72	0.247

Area	Domain	Metal	Ellipsoid Angles			Nugget	Structure 1 Ranges (m)			C1	Structure 2 Ranges (m)			C2
			1 (X)	2 (Y)	3 (Z)		1	2	3		1	2	3	
SE	E-W	Cu	-152	93	27	0.278	8.1	15.1	59.1	0.348	1,100	380	141	0.374
SEHW	NW-SE	Au	-58	-29	-52	0.032	4.1	23.6	4	0.900	600	100	30	0.068
SEHW	NW-SE	Ag	-58	-41	92	0.321	70.2	54.3	9.8	0.396	696	357	52	0.290
SEHW	NW-SE	Cu	-45	59	6	0.213	24.2	7.4	71.3	0.468	1,342	262	396	0.319
SEFW	NW-SE	Au	-58	-29	-52	0.032	4.1	23.6	4	0.900	600	100	30	0.068
SEFW	NW-SE	Ag	-58	-41	92	0.321	70.2	54.3	9.8	0.396	696	357	52	0.290
SEFW	NW-SE	Cu	-45	59	6	0.213	24.2	7.4	71.3	0.468	1,342	262	396	0.319
Sona Hill Deposit														
Sona Hill	Background	Au	-11	34	-32	0.018	5	19.1	3.3	0.837	116	86	49	0.146
Sona Hill	Background	Ag	-11	34	-32	0.018	5	19.1	3.3	0.837	116	86	49	0.146
Sona Hill	Background	Cu	-11	34	-32	0.018	5	19.1	3.3	0.837	116	86	49	0.146
Sona Hill	NW-SE	Au	-35	-102	12	0.000	1.1	16.3	10.1	0.423	13.5	29.3	2.7	0.577
Sona Hill	NW-SE	Ag	-35	-102	12	0.000	1.1	16.3	10.1	0.423	13.5	29.3	2.7	0.577
Sona Hill	NW-SE	Cu	-35	-102	12	0.000	1.1	16.3	10.1	0.423	13.5	29.3	2.7	0.577
Sona Hill	E-W	Au	-35	-102	12	0.000	1.1	16.3	10.1	0.423	13.5	29.3	2.7	0.577
Sona Hill	E-W	Ag	-35	-102	12	0.000	1.1	16.3	10.1	0.423	13.5	29.3	2.7	0.577
Sona Hill	E-W	Cu	-35	-102	12	0.000	1.1	16.3	10.1	0.423	13.5	29.3	2.7	0.577

14.6 Estimation of Non Payables

Non payable elements were estimated using the NN and ID2 interpolation methods. They included silver, arsenic, cadmium, calcium, copper, lead, and sulphur.

14.7 Block Model Validation

The block model validation process included visual comparisons between block estimates and composite grades in plan and section, local versus global estimates for NN, ID2, ID3, and OK, as well as swath plots. In addition, block estimates were visually compared to the drill hole composite data in all Domains and corresponding zones to ensure agreement. No material grade bias issues were identified, and the block model grades compared well to the composite data.

14.7.1 Visual Block Model Validation

The validation of the interpolated block model was performed by using visual assessments and validation plots of block grades versus capped assay grades. The review demonstrated a good comparison between local block estimates and nearby assays and composites without excessive smoothing in the block model. Figure 14-19 through Figure 14-23 provide visual comparisons, displaying raw gold assay grades versus block model grades.

More visual block model validation images, including gold, copper, and silver, are available in Appendix C.

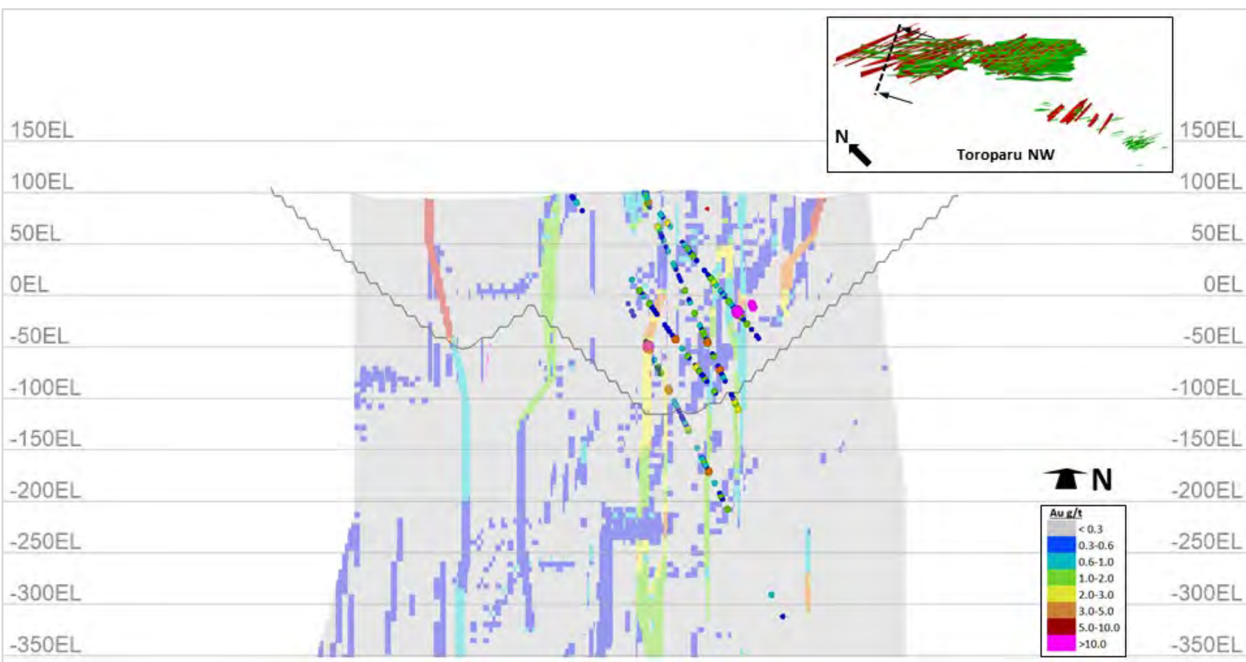


Figure 14-19: Block model validation, Toroparu Deposit, NW Area

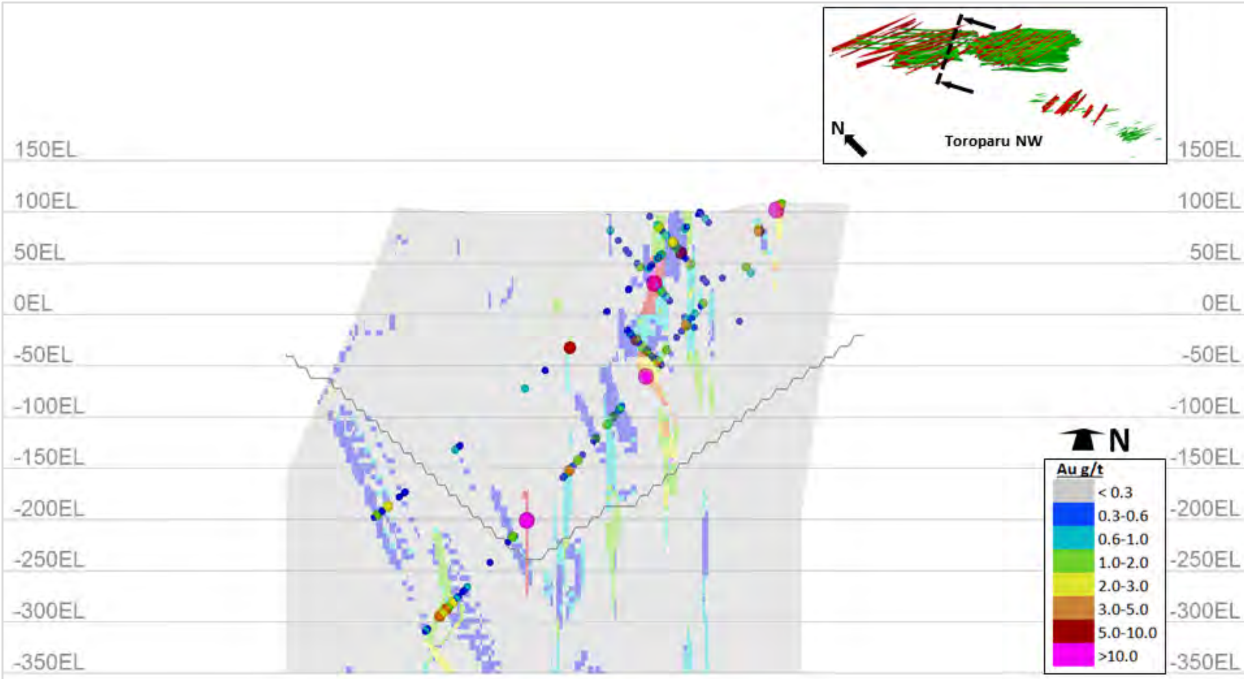


Figure 14-20: Block model validation, Toroparu Deposit, Main Area

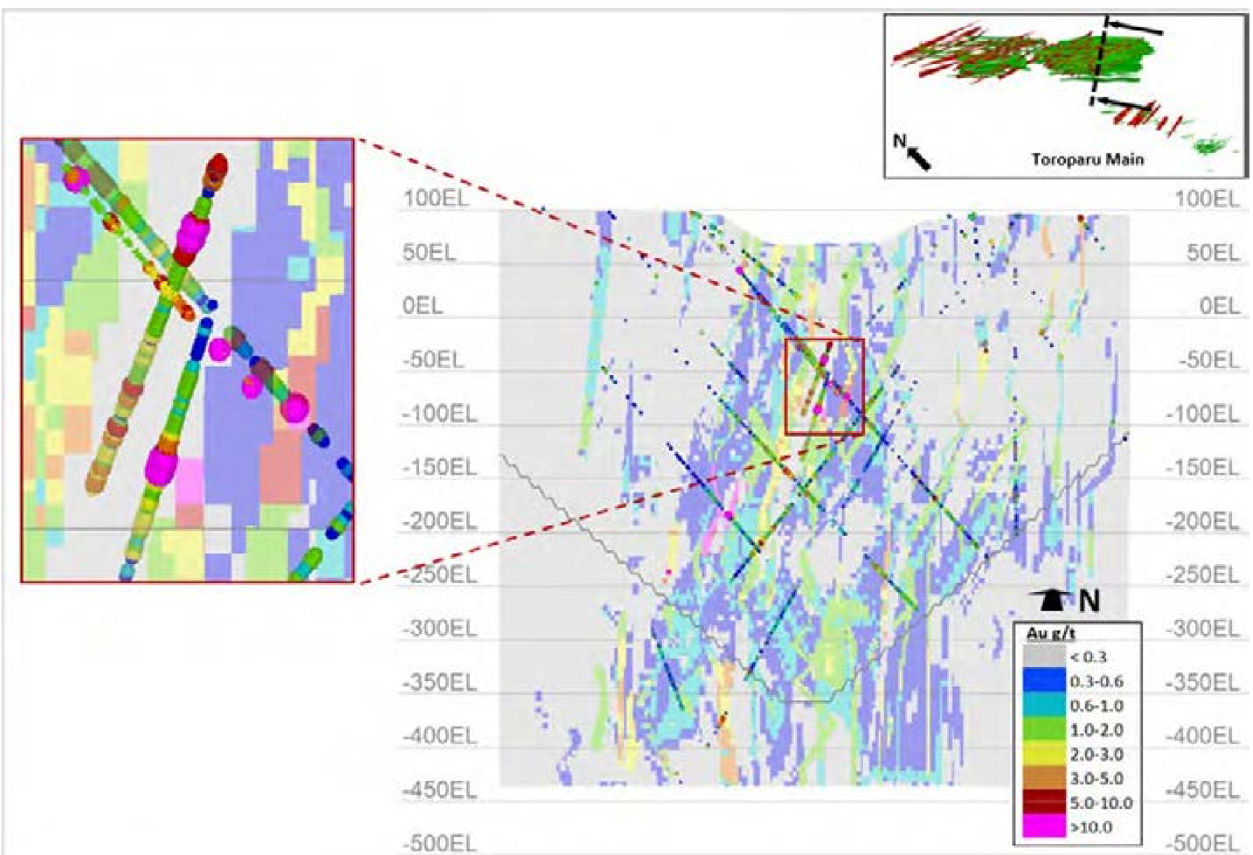


Figure 14-21: Block model validation, Toroparu Deposit, Main Area

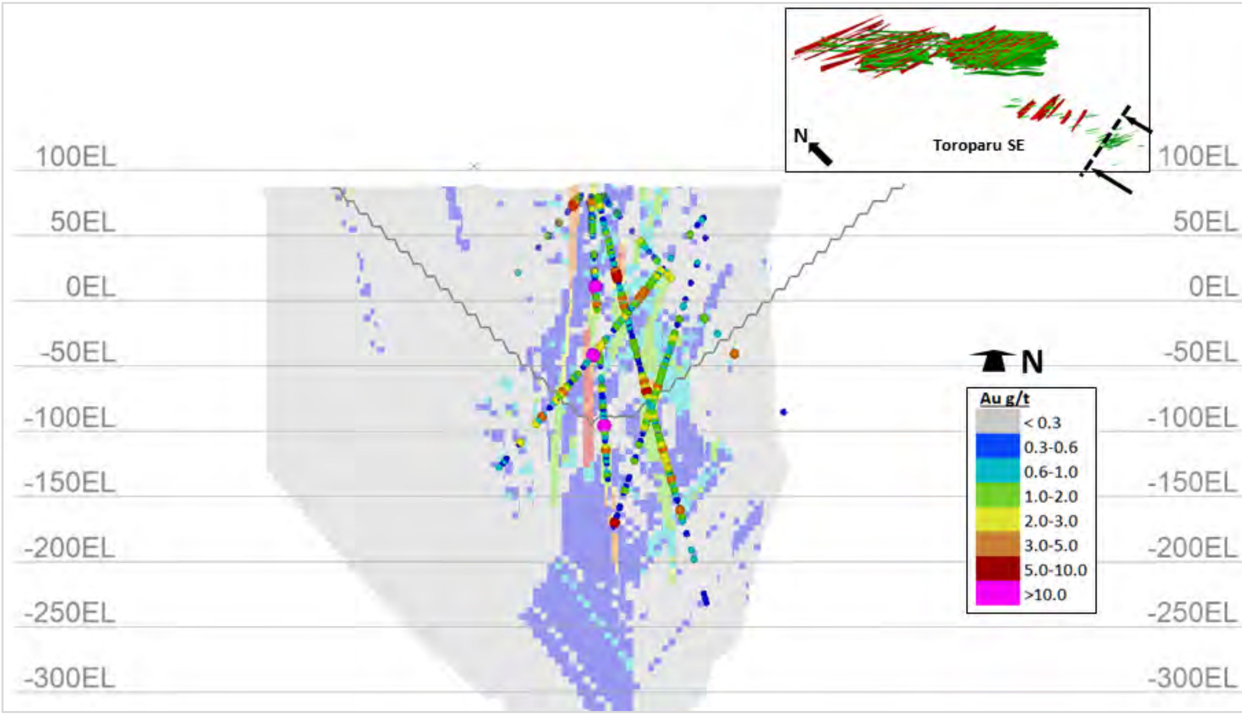


Figure 14-22: Block model validation, Toroparu Deposit, SE Area

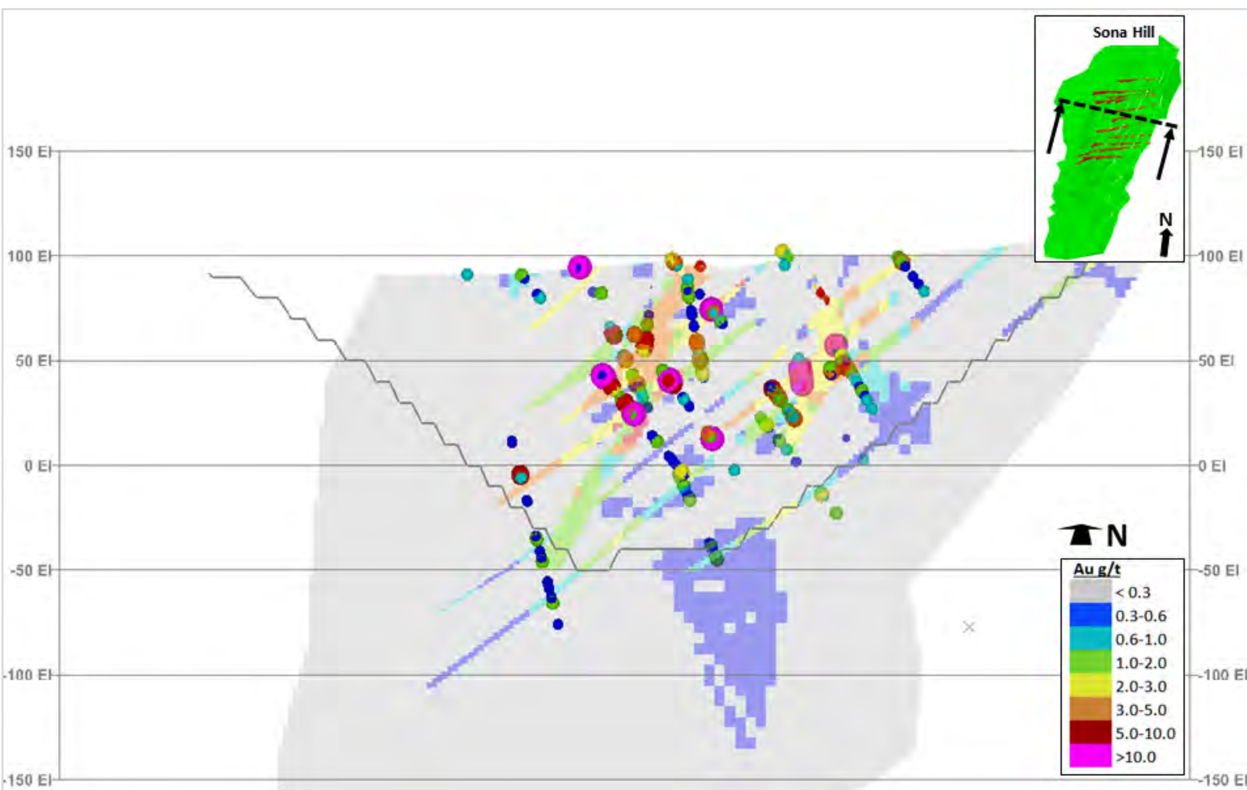


Figure 14-23: Block model validation, Sona Hill Deposit

14.7.2 Swath Plots

A swath plot is a graphical representation of grade distribution derived by a series of sectional "swaths" throughout the deposits. Swath plots were generated for gold from slices throughout each block model. They compare the block model grades for NN, ID2, ID3, and OK to the drill hole composite grades to evaluate any potential local grade bias. A review of the swath plots did not identify bias in the model that is material to the 2021 Mineral Resource Estimate, as there was a strong overall correlation between the block model OK grade and the capped composites used in the 2021 Mineral Resource Estimate. Figure 14-24, Figure 14-25 and Figure 14-26 provide the Toroparu Deposit swath plots. Figure 14-27 and Figure 14-28 provide the Sona Hill Deposit swath plots. For these figures, the composite grade (S_AUCAP) is compared across swaths with the four gold estimation grades from the block model.

Fields include (gold and silver grade units are g/t, copper grade units are percentage):

- M_TONNES: Block model tonnage
- NRECORDS: Number of records
- S_AUCAP: composite capped gold grade
- M_AUOK: Block model estimated gold grade, OK
- M_AUID2: Block model estimated gold grade, ID2
- M_AUID3: Block model estimated gold grade, ID3
- M_AUNN: Block model estimated gold grade, NN

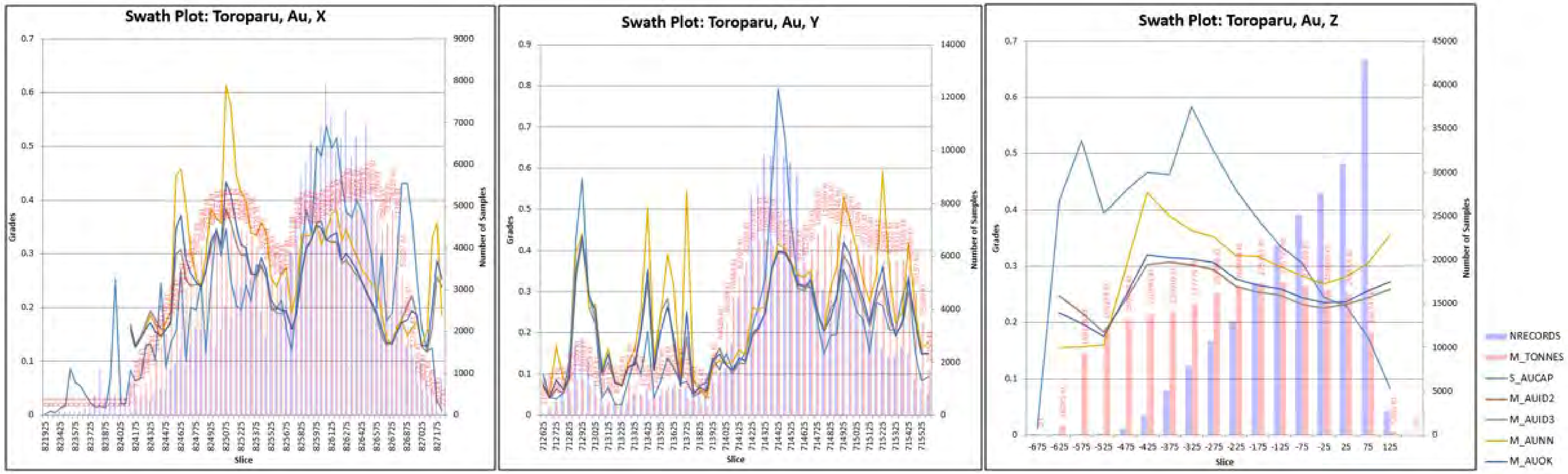


Figure 14-24: Swath plots, Toroparu Deposit, gold

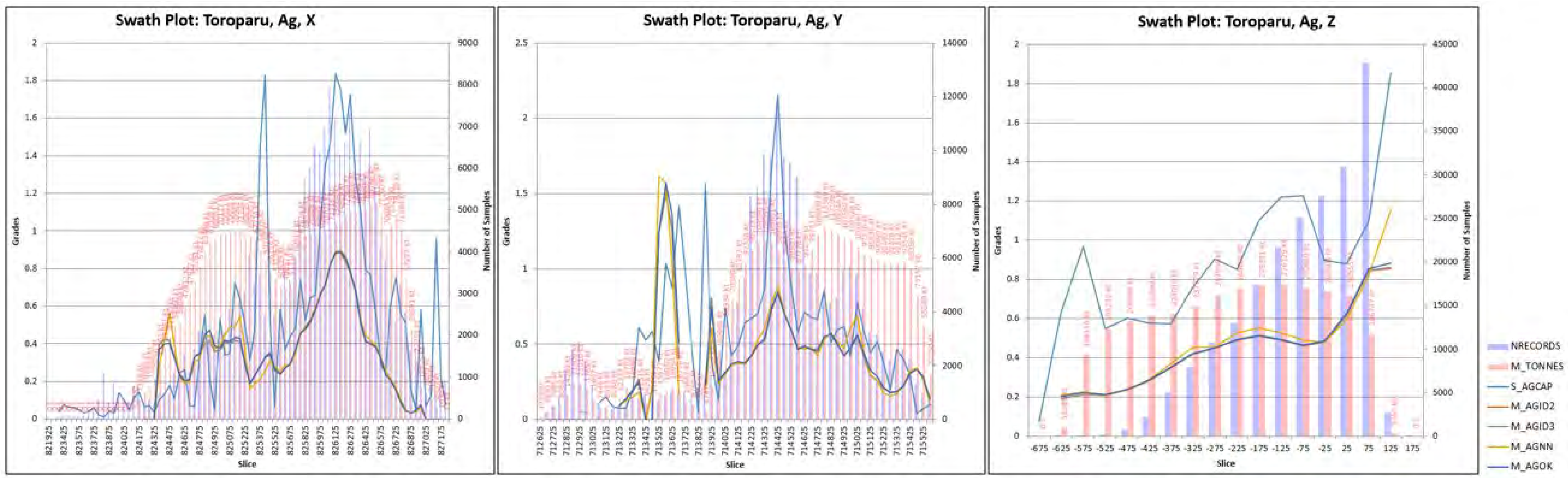


Figure 14-25: Swath plots, Toroparu Deposit, gold

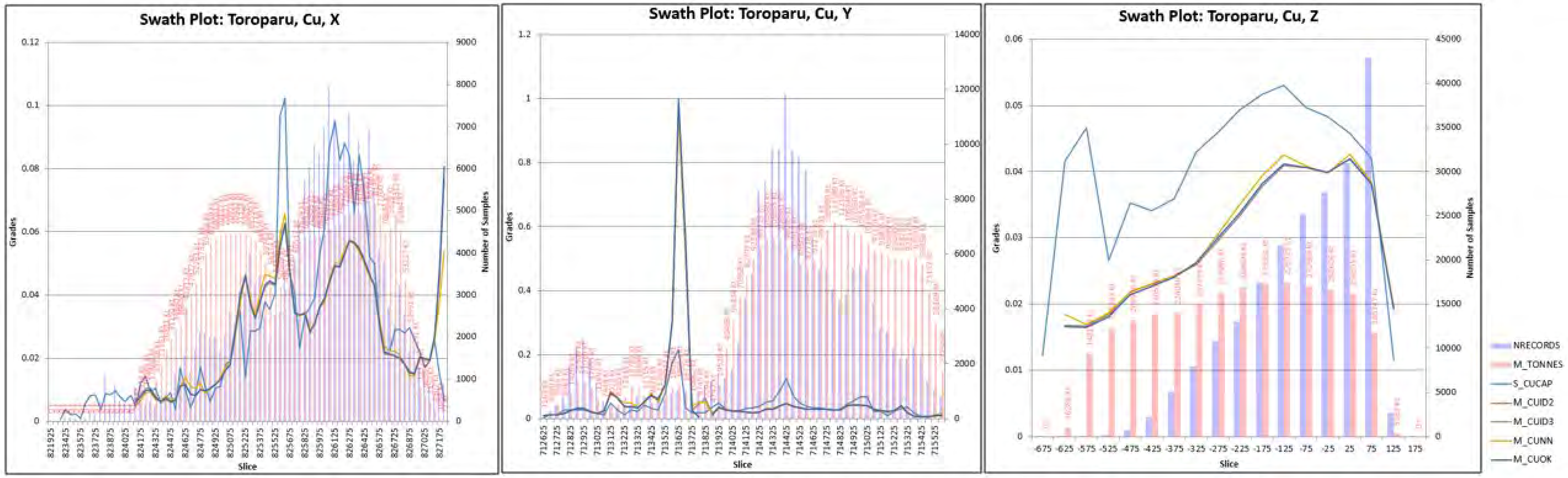


Figure 14-22: Swath plots, Toroparu Deposit, copper

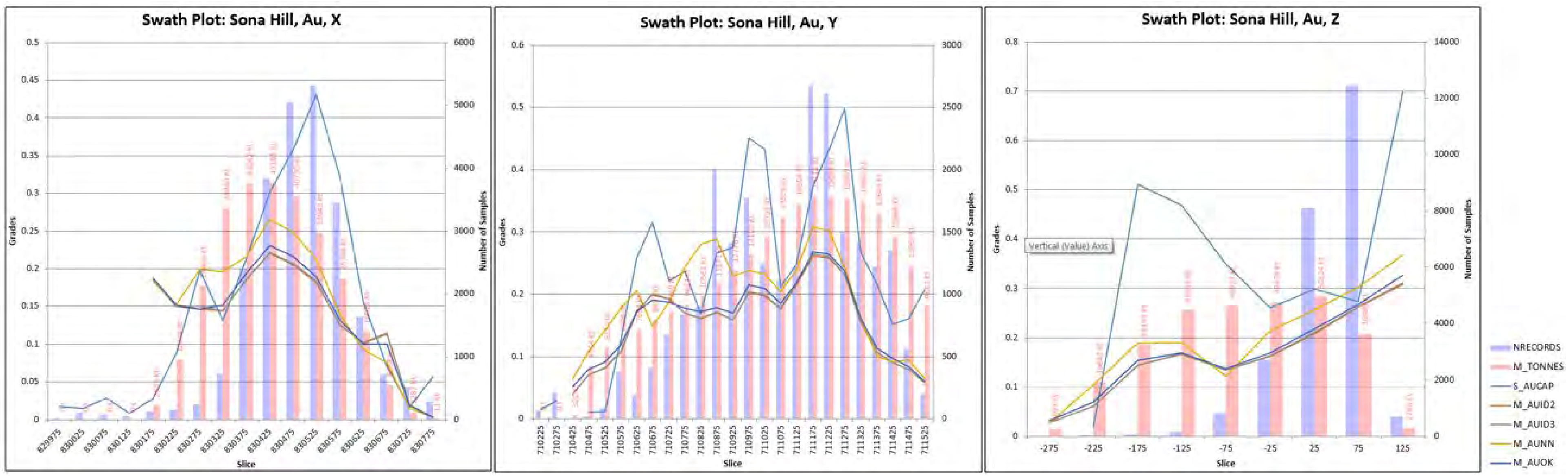


Figure 14-27: Swath plots, Sona Hill Deposit, gold

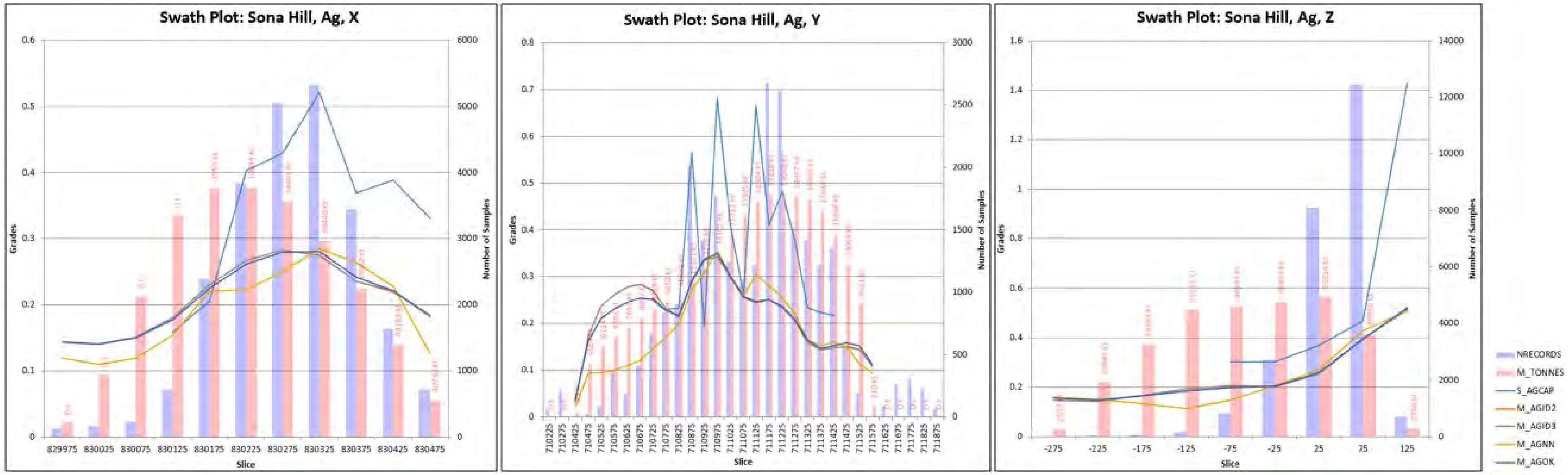


Figure 14-28: Swath plots, Sona Hill Deposit, silver

14.8 Interpolation Comparison

Estimation was completed using NN, ID2, ID3, and OK interpolation methods. The results are presented in Table 14-29. This includes all open pit material that has been classified as Measured, Indicated, and Inferred.

Table 14-29: Interpolation Comparison

Toroparu Deposit Open Pit															
Area	Rock Type	Classification	Cut-off Au g/t	Au g/t OK	Au g/t ID2	Au g/t ID3	Au g/t NN	Ag g/t OK	Ag g/t ID2	Ag g/t ID3	Ag g/t NN	Cu % OK	Cu % ID2	Cu % ID3	Cu % NN
Main/NW	Saprolite	Measured	0.33	1.25	1.13	1.13	1.27	1.48	1.47	1.48	1.55	0.12	0.12	0.12	0.12
		Indicated	0.33	1.77	1.52	1.52	1.81	1.22	1.21	1.22	1.30	0.11	0.12	0.11	0.11
		Inferred	0.33	2.32	2.38	2.40	2.39	1.02	1.10	1.15	1.17	0.08	0.09	0.09	0.09
	Fresh	Measured	0.57	2.62	2.26	2.28	3.23	1.76	1.74	1.75	1.98	0.18	0.19	0.18	0.18
		Indicated	0.57	4.42	3.60	3.61	6.18	0.88	0.85	0.86	1.17	0.10	0.11	0.10	0.10
		Inferred	0.57	3.09	2.85	2.87	3.34	0.61	0.61	0.61	0.80	0.08	0.11	0.09	0.09
SE	Saprolite	Measured	0.33	1.02	0.94	0.94	1.06	n/a	n/a	n/a	n/a	0.05	0.05	0.05	0.05
		Indicated	0.33	0.99	0.90	0.90	1.03	n/a	n/a	n/a	n/a	0.06	0.07	0.06	0.06
		Inferred	0.33	1.83	1.53	1.45	1.42	n/a	n/a	n/a	n/a	0.05	0.03	0.05	0.05
	Fresh	Measured	0.57	1.43	1.35	1.36	1.58	n/a	n/a	n/a	n/a	0.04	0.04	0.04	0.04
		Indicated	0.57	1.51	1.40	1.41	1.80	n/a	n/a	n/a	n/a	0.05	0.05	0.05	0.05
		Inferred	0.57	1.47	1.29	1.25	1.30	n/a	n/a	n/a	n/a	0.03	0.04	0.04	0.04
Sona Hill Deposit Open Pit															
Sona Hill	Saprolite	Measured	0.40	1.68	1.62	1.63	1.96	1.15	1.14	1.16	1.34	n/a	n/a	n/a	n/a
		Indicated	0.40	1.42	1.34	1.35	1.62	0.96	0.97	1.00	1.29	n/a	n/a	n/a	n/a
		Inferred	0.40	1.00	0.98	1.00	1.09	0.84	0.83	0.84	1.23	n/a	n/a	n/a	n/a
	Fresh	Measured	0.64	2.35	2.18	2.20	2.75	1.31	1.28	1.28	1.48	n/a	n/a	n/a	n/a
		Indicated	0.64	2.21	2.16	2.21	2.80	1.13	1.12	1.13	1.38	n/a	n/a	n/a	n/a
		Inferred	0.64	1.81	1.76	1.77	2.08	0.99	1.05	1.07	1.52	n/a	n/a	n/a	n/a

14.9 Mineral Resource Classification

The Mineral Resource was classified in accordance with the 2014 CIM Definition Standards and 2019 CIM Best Practice Guidelines. Mineral Resource classifications or "categories" were assigned to regions of the block model based on the QP's confidence and judgment related to geological understanding, continuity of mineralization in conjunction with data quality, spatial continuity based on variography, estimation pass, data density, and block model representativeness, specifically assay spacing and abundance, kriging variance, and search volume block estimation assignment.

For the Toroparu Deposit, the classification was initially applied from the estimation pass. Blocks populated in pass 1 were classified as Measured, blocks populated in pass 2 were classified as Indicated, and blocks populated in pass 3 were classified as Inferred. Subsequently, the block model was analyzed, and it was determined that classification adjustments were required depending on the drilling density required to support an underground or an open pit resource; blocks in the first, second, and third pass that display a relatively high kriging variance were downgraded to a lower classification. The Toroparu Deposit classification can be seen in Figure 14-29 through Figure 14-28.

For the Sona Hill Deposit, classification was applied directly from the estimation pass. Blocks populated in pass 1 were classified as Measured, blocks populated in pass 2 were classified as Indicated, and blocks populated in pass 3 were classified as Inferred. The Toroparu Deposit classification can be seen in Figure 14-29 and Figure 14-30. More images are available in Appendix D.

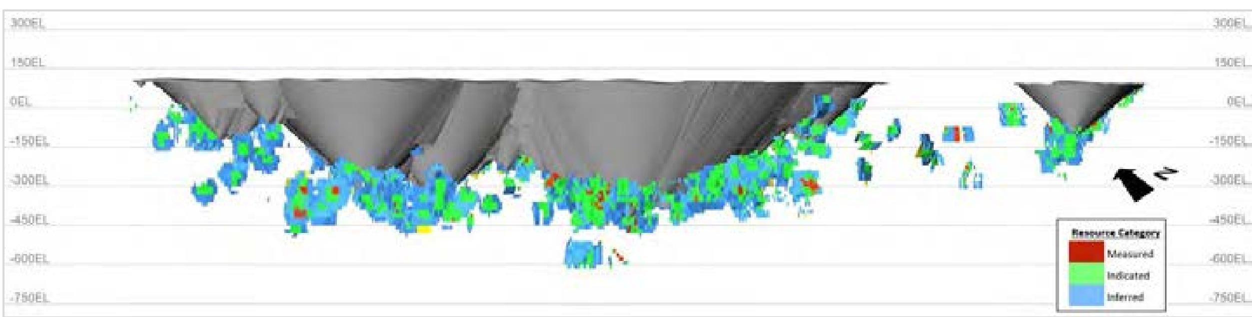


Figure 14-23: Toroparu Deposit underground classification, long section

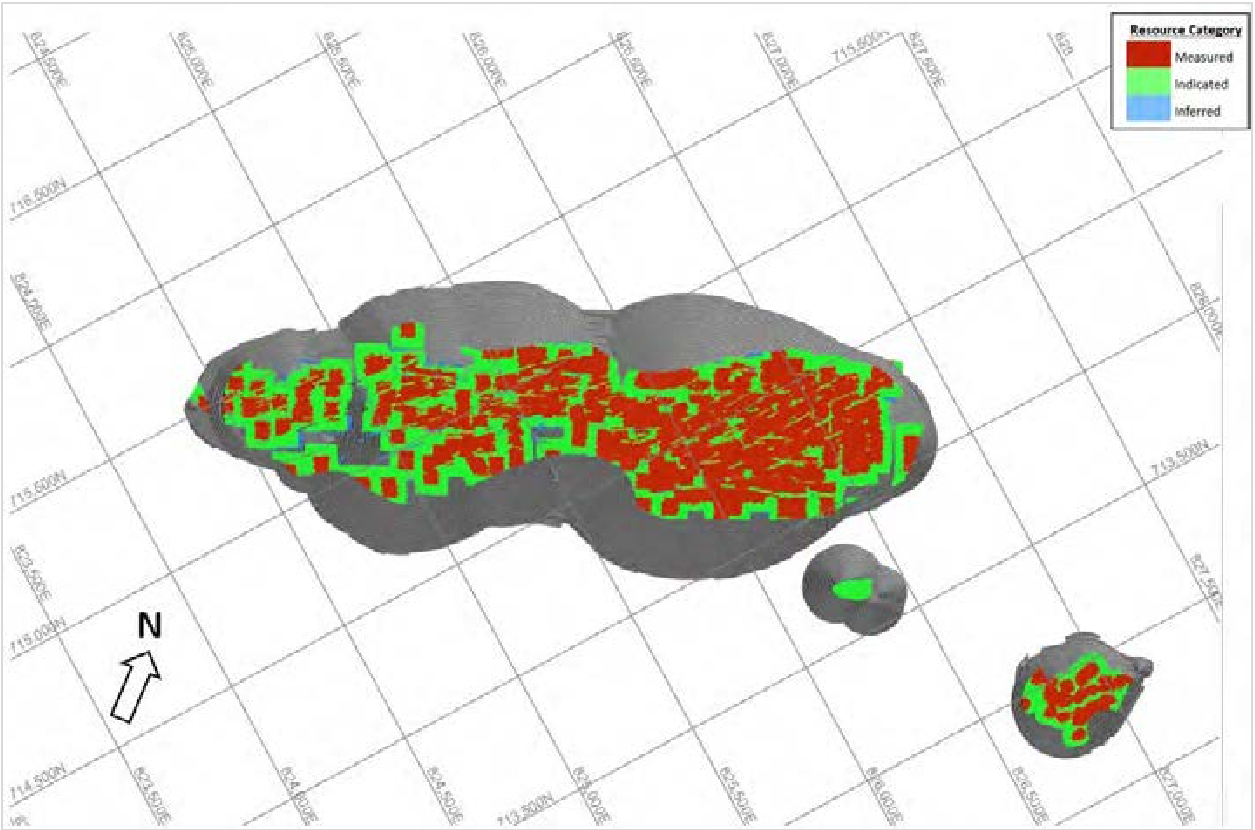


Figure 14-24: Toroparu Deposit, NW, and Main Area open pit classification, plan section

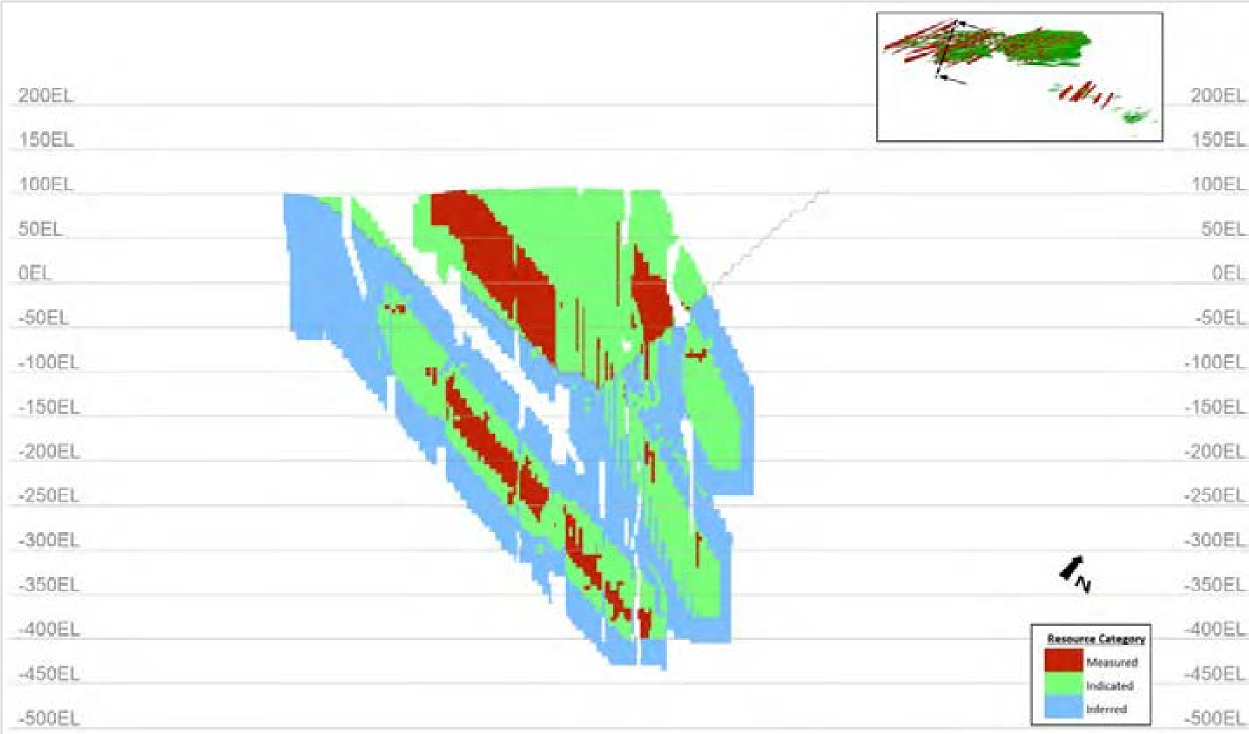


Figure 14-25: Toroparu Deposit, NW Area combined open pit and underground classification, cross-section

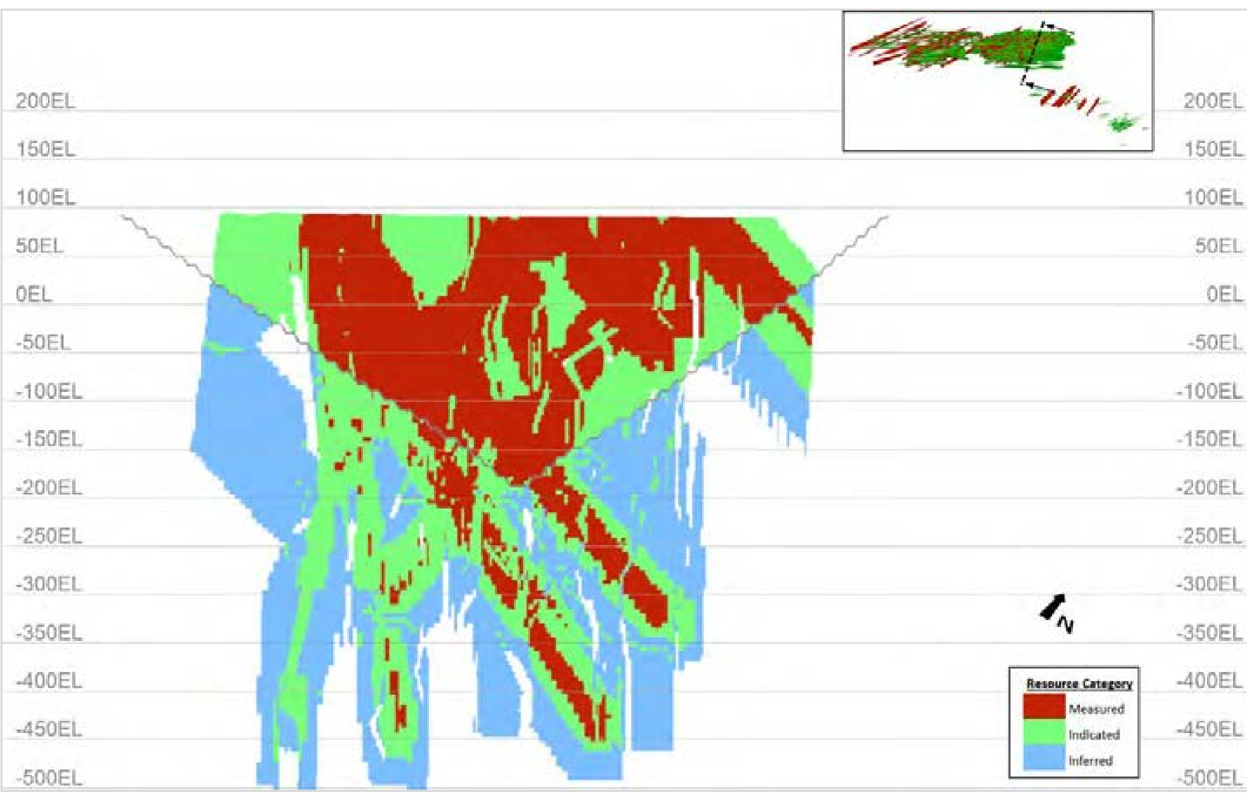


Figure 14-26: Toroparu Deposit, Main Area combined open pit and underground classification, cross-section

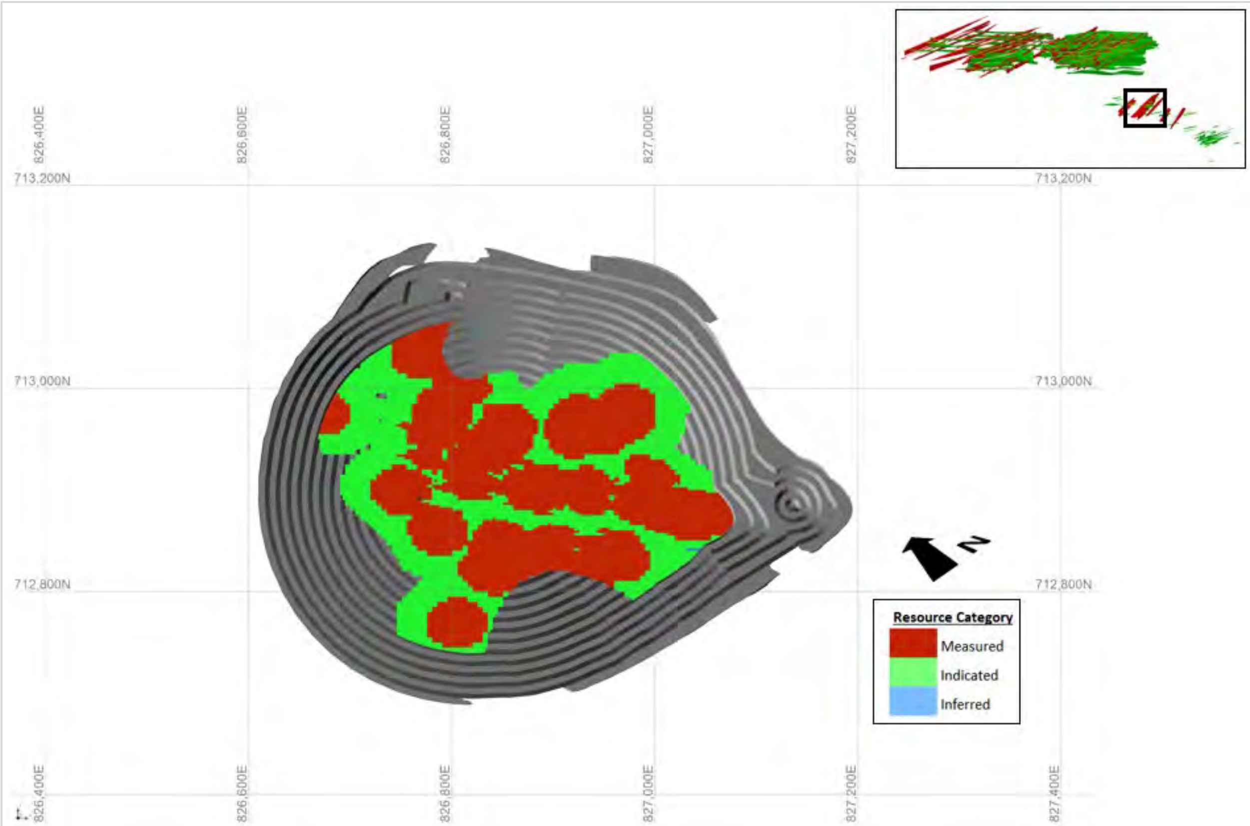


Figure 14-27: Toroparu Deposit, SE Area open pit classification, plan view

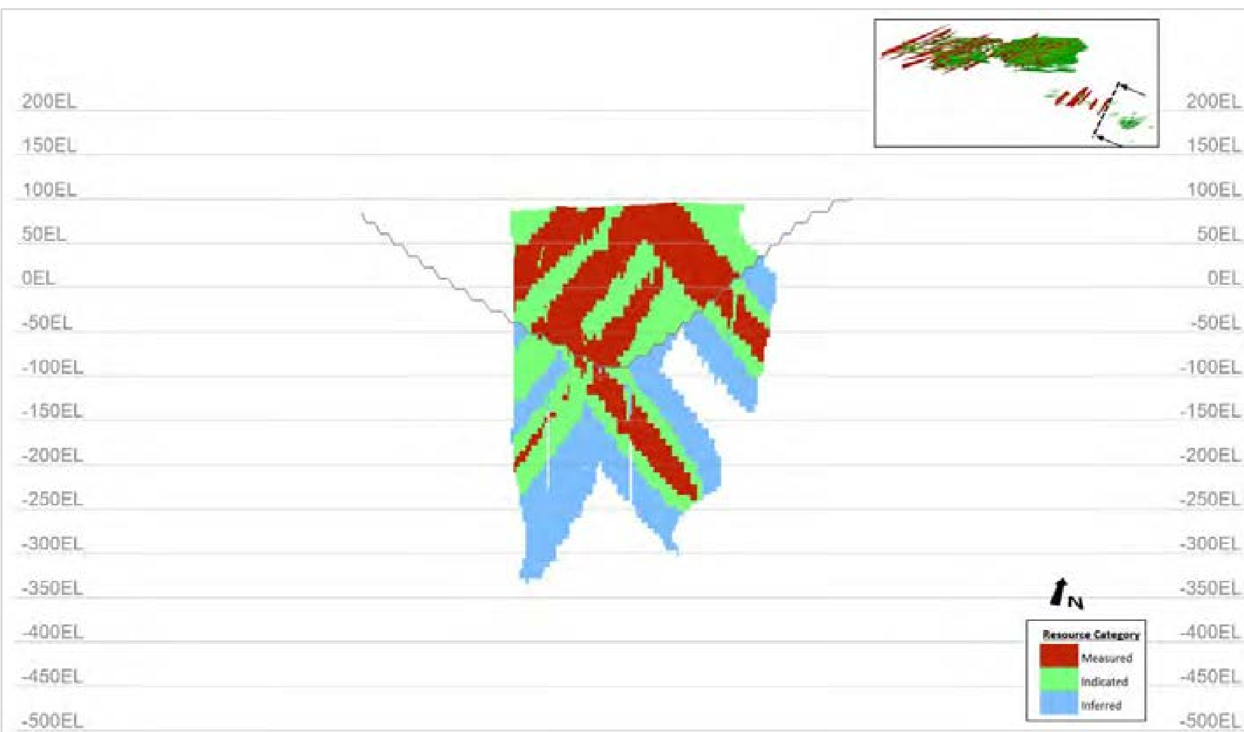


Figure 14-28: Toroparu Deposit, SE Area combined open pit and underground classification, cross-section

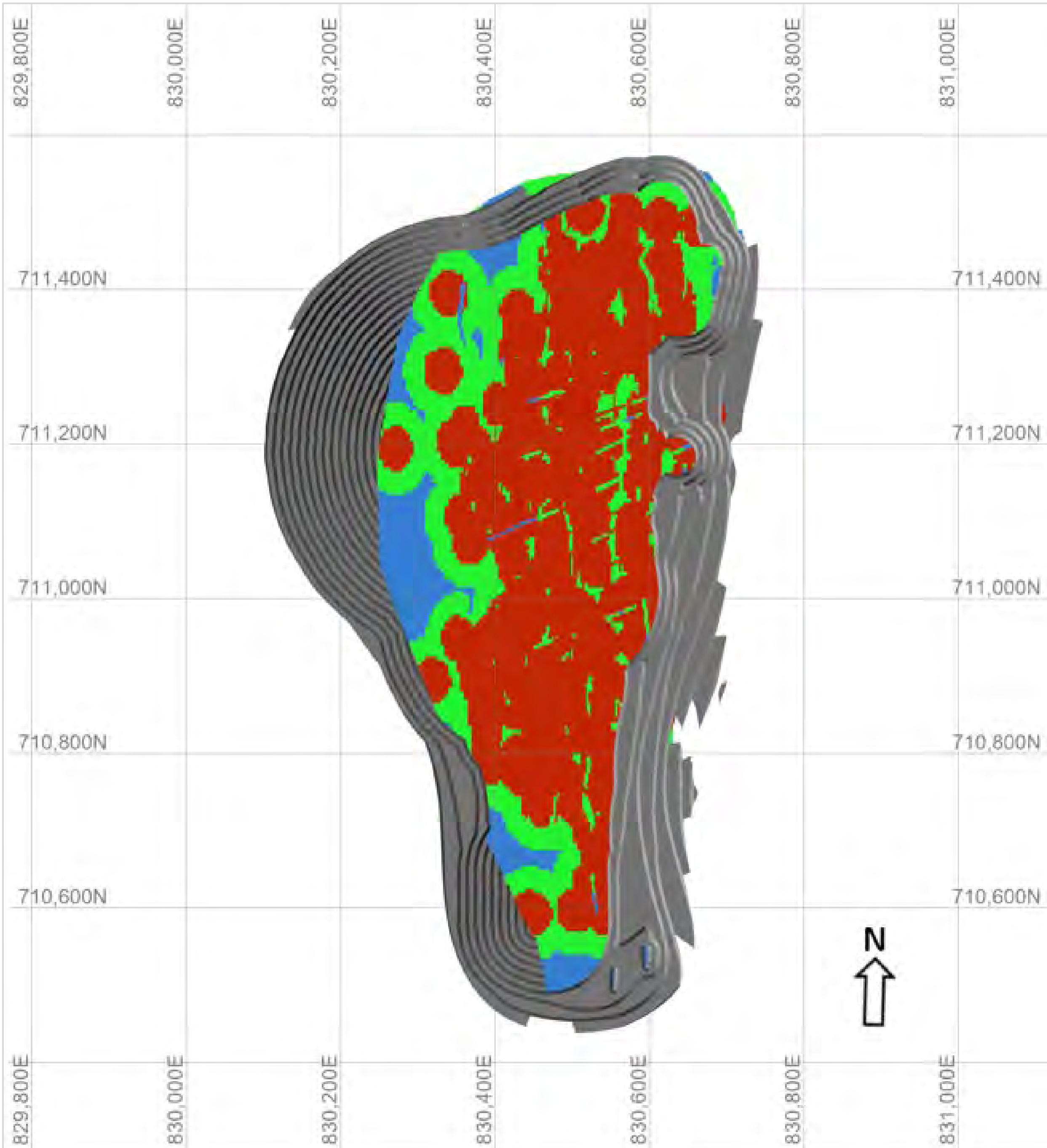


Figure 14-29: Sona Hill Deposit open pit classification, plan view

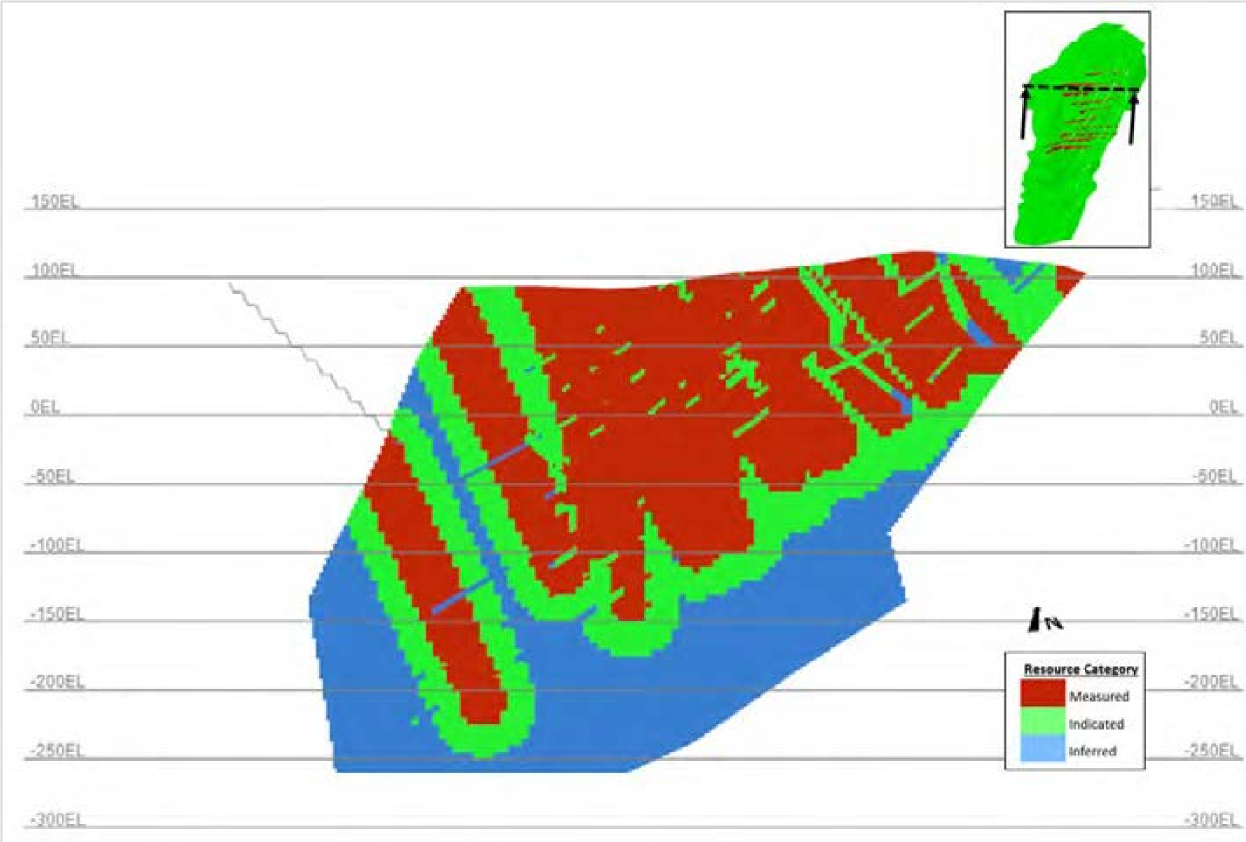


Figure 14-30: Sona Hill Deposit classification, cross-section

14.10 Reasonable Prospects of Eventual Economic Extraction

14.10.1 Underground

For the Underground Mineral Resource Estimate (Table 14-30), representational minable shapes were created using MSO in Datamine™ Studio underground 2.5.40.0 software to determine physical limits for an MSO constrained Mineral Resource Estimate. The Underground Mineral Resource cut-off grade is estimated to be 1.80 g/t gold. The parameters used to calculate the cut-off grade and for MSO parameters are shown in Table 4. A mining recovery of 95% and 16% mining dilution was applied for resource cut-off calculation purposes. The Underground Mineral Resource Estimate comprises all material found within MSO wireframes generated at a cut-off of 1.80 g/t gold, including material below cut-off.

Table 14-30: Underground Mineral Resource Cut-off Grade and MSO Parameters

Parameter	Value
Currency Used for Evaluation	US$
Underground Mining Cost includes assumptions for operating labour, consumables, power, surface hauling	25.00$/t processed
Underground Support Cost Includes assumptions for infill diamond drilling, equipment maintenance, technical services	11.00$/t processed

Parameter	Value
Process Cost includes assumptions for milling, tailings, water treatment	15.50$/ t processed
G&A Cost Includes assumptions for camp, off site materials transportation	6.00$/t processed
Operating Cost Marginal Allowance 10% Marginal allowance to cover a portion of capital costs	5.80$/t processed
Selling Cost includes doré transportation, refining, and 8% government royalty	133$/t.oz.
% Payable	99.95%
Metal Price	1,630 US$/t.oz.
Mining Dilution	15%
Mining Recovery	95%
Process Recovery	87.5%
Production Rate Assumption	3,500 tonne per day
MSO Shape Parameters	Height: 30 m, 2 Sub shapes possible @15 m Length: 25 m, 4 Sub shapes possible @6.25 m Width: Variable 2 m to 25 m Minimum Dip: 50 °

14.10.2 Open Pit

For the open pit Mineral Resource Estimate (Table 1), a pit limit analysis was undertaken using the Lerchs-Grossmann ("LG") algorithm in Geovia's Whittle™ 4.7 software to determine physical limits for a pit shell constrained Mineral Resource Estimate. The parameters used to generate a pit shell are shown in Table 14-31 and Table 14-32.

Table 14-31: Physical Pit Limit Analysis Parameters, Toroparu Deposit, Main, NW, and SE Areas

Parameter	Value		
Currency Used for Evaluation	US$		
Block Size	Parent Block Model size 5 m x 5 m x 5 m		
Mining Loss Assumption Mining Dilution Assumption	0% 0%, above the inherent dilution in the parent block size		
	Saprolite	**Fresh Rock**	
Overall Slope Angle	30°	45°	
Mining Cost	2.30$/t mined	2.30$/t mined	
Process Cost includes Milling and Tailings	**CIL – Saprolite**	**CIL – Fresh**	**Flotation – Fresh**
	13.50$/t processed	21.45$/t processed	15.50$/t processed
G&A	5.75 $/T processed	5.75 $/T processed	5.75 $/T processed
Sustaining Capex, Closure	0.20 $/T processed	0.20 $/T processed	0.20 $/T processed
Process Recovery	97.5% Au	92.8% Au	80.0% Au 88% Cu 72.0% Ag
Selling Cost includes royalty	8% Au	8% Au	8% Au 1.5% Cu
Metal Price	1,630 $/oz Au 3.13 $/lb Cu		
Resources Used to Generate Pit Shell	Measured + Indicated + Inferred Resources		
Pit Shell Selection	Revenue Factor (RF) 0.75 Toroparu Main/NW/SE		

Table 14-32: Physical Pit Limit Analysis Parameters, Sona Hill Deposit

Parameter	Value	
Currency Used for Evaluation	US$	
Block Size	Parent Block Model size 2.5 m x 2.55 m x 5 m	
Mining Loss Assumption Mining Dilution Assumption	0% 0%, above the inherent dilution in the parent block size	
	Saprolite	**Fresh Rock**
Overall Slope Angle	30°	45°
Mining Cost	2.30$/t mined	2.30$/t mined
Process Cost Includes Milling and Tailings	**CIL – Saprolite**	**CIL – Fresh**
	13.50$/t processed	21.45$/t processed
G&A	5.75 $/T processed	5.75 $/T processed
Sustaining Capex, Closure	0.20	0.20
Process Recovery	98% Au	83% Au
Selling Cost includes royalty	8% Au	8% Au
Metal Price	1,630 $/oz Au	
Resources Used to Generate Pit Shell	Measured + Indicated + Inferred Resources	
Pit Shell Selection	RF 1.00 (Sona Hill)	

The milling cut-off grade is used to classify the material contained within the pit shell limits as open pit resource material. This break-even cut-off grade is calculated to cover the Process and Selling Costs using the parameters listed in Table 5 and Table 6. A mining recovery of 100% and 0% mining dilution was applied for resource cut-off calculation purposes.

14.11 Mineral Resource Estimate

The Mineral Resources were classified using the 2014 CIM Definition Standards and the 2019 CIM Best Practice Guidelines. The Mineral Resource Estimate has an effective date of November 1, 2021.

The maiden underground resource estimate defines high-grade mineralization within multiple discreet northwest and east-west oriented structures that intersect in a repeatable pattern for at least 4 km along the northwest strike, approximately 400 m to 450 m wide and over 450 m to 500 m in depth. The Mineral Resource Estimate includes both resources amenable to underground mining methods and the mining of a starter open pit within the existing 3 km strike Toroparu Deposit and the mining of the Sona Hill Deposit and SE Area open pits. The Optimized Mineral Resource is effective from November 1, 2021, supersedes all previous Mineral Resource Estimates and Technical Reports filed by the Company. The Mineral Resource Estimate was optimized from previous estimates based on an updated understanding of the geologic structural control over gold mineralization to define a lower volume, higher-grade core resource that can be mined using open pit and underground mining methods.

The Toroparu Deposit Mineral Resource Estimate presented in

Table 14-16 is based on validated results of 893 diamond drill holes totalling 215,346.6 m and three trenches comprised of 655.3 m completed between 2006 and the effective date. The updated resource estimate includes the 2020-2021 drill program which comprised a total of 20,750 m in 114 drill holes. The Sona Hill Deposit Mineral Resource Estimate, also presented in

Table 14-16, is based on validated results of 181 diamond drill holes for a total of 20,850 m completed between 2015 and 2018.

Table 14-33: Mineral Resource Statement for the Toroparu Project

Deposit	Area	Resource Category	Type	Tonnes ('000s)	Au (g/t)	Au oz ('000s)	Cu (%)	Cu lb ('000s)	Ag (g/t)	Ag oz ('000s)
Toroparu	**Main/NW**	Measured	Open pit	98,070	1.21	3,809	0.110	238,112	1.19	3,743
		Indicated		62,531	1.56	3,133	0.100	137,557	0.91	1,828
Toroparu	**SE**	Measured	Open pit	5,121	1.16	190	0.043	4,826	n/a	n/a
		Indicated		2,403	1.14	88	0.052	2,763	n/a	n/a
Sona Hill	**Sona Hill**	Measured	Open pit	6,958	1.85	413	0.008	1,241	1.07	239
		Indicated		4,180	1.66	223	0.008	700	0.85	115
Toroparu	**Main/NW**	Measured	Underground	727	2.84	66	0.072	1,151	0.47	11
		Indicated		4,978	3.21	514	0.091	9,937	0.41	66
Total Measured				**110,877**	**1.26**	**4,479**	**0.100**	**245,330**	**1.12**	**3,993**
Total Indicated				**74,092**	**1.66**	**3,958**	**0.092**	**150,957**	**0.84**	**2,009**
Total Measured & Indicated				**184,969**	**1.42**	**8,437**	**0.097**	**396,286**	**1.01**	**6,002**
Toroparu	**Main/NW**	Inferred	Open Pit	4,018	1.58	204	0.080	7,118	0.66	85
Toroparu	**SE**	Inferred	Open Pit	9	1.67	1	0.040	8	n/a	n/a
Sona Hill	**Sona Hill**	Inferred	Open Pit	1,365	1.28	56	0.006	179	0.54	24
Toroparu	**Main/NW/SE**	Inferred	Underground	8,403	3.53	953	0.091	16,884	0.25	68
Total Inferred				**13,796**	**2.74**	**1,213**	**0.08**	**24,189**	**0.40**	**177**

Mineral Resource Estimate Notes

1. Combined Open Pit and Underground Mineral Resources were prepared in accordance with NI 43-101 and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. This estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
2. Underground and Open Pit Mineral Resources are based on a gold price of $1,630/oz. This gold price is the three-year tailing average as of September 30, 2021.
3. Open Pit Mineral Resources comprise the material contained within various Lerchs-Grossmann pit shells at various revenue factors. These revenue factors are as follows: Main/Southeast/NW Zone @ 0.75 revenue factor and Sona Hill @ 1.00 revenue factor. The gold cut-off applied to Open Pit Mineral Resources within the selected pit shells was 0.40 g/t.
4. Underground Mineral Resources comprise all material found within MSO wireframes generated at a cut-off of 1.8 g/t gold including material below cut-off.
5. Silver values are not reported for the SE Open Pit Ag contained metal values reported will not equal A tonnes X grade conversion calculation.
6. Assays were variably capped on a wireframe-by-wireframe basis.
7. Specific Gravity was applied using weighted averages to each individual lithology type.
8. Mineral Resource effective date November 1, 2021.
9. All figures are rounded to reflect the relative accuracy of the estimates and totals may not add correctly.
10. Excludes unclassified mineralization located within mined out areas.
11. Reported from within a mineralization envelope accounting for mineral continuity.

14.11.1 Underground Resource Estimate

Table 14-34 provides the Underground Mineral Resource Estimate by classification of the Toroparu Deposit.

Table 14-34: Underground Mineral Resource Estimate by Classification

Location	Resource Category	Tonnes (‘000s)	Au (g/t)	Au oz (‘000s)	Cu (%)	Cu lb (‘000s)	Ag (g/t)	Ag oz (‘000s)
Main/NW	Measured & Indicated	5,705	3.16	580	0.088	11,088	0.42	77
	Inferred	8,403	3.53	953	0.091	16,884	0.25	68

The intersection of NW-SE/E-W oriented structures supports underground mining methods. To demonstrate reasonable prospects of eventual economic extraction by underground mining methods, Nordmin designed representational minable shapes for the mineralization using industry standard MSO in Datamine™ Studio underground 2.5.40.0. The software determined the physical limits for an MSO constrained Underground Mineral Resource Estimate (Mineral Resource Estimate). A mining recovery of 95% and 16% mining dilution was applied for resource cut-off calculation purposes. The Underground

Mineral Resource Estimate comprises all material found within MSO wireframes generated at a cut-off of 1.8 g/t gold, including material below cut-off.

The long section in Figure 14-31 illustrates the outline of the underground minable shapes within the Mineral Resource Estimate above the cut-off grade for the Toroparu Deposit. The underground resource excludes a 30 m pillar below the bottom of the proposed open pits. The underground resource extends over a 3,000 m strike length x 400 m width x 450-500 m depth. The extent of mineralization is open to the northwest and at depth as existing drilling data has not defined these boundaries.

Figure 14-32 provides cross-section views of the gold mineralization contained in Main and NW Areas of the Toroparu Deposit and an outline of the Starter Open Pit and Saprolite/Fresh Rock Boundary.

Figure 14-33 provides cross-section views of the gold mineralization contained in the Main and NW Areas of the Toroparu Deposit.

Figure 14-34 provides the cross-section views of the gold mineralization contained both within and extending below the open pit boundaries as well as the saprolite/fresh rock boundary in the SE Area and Sona Hill Deposit.

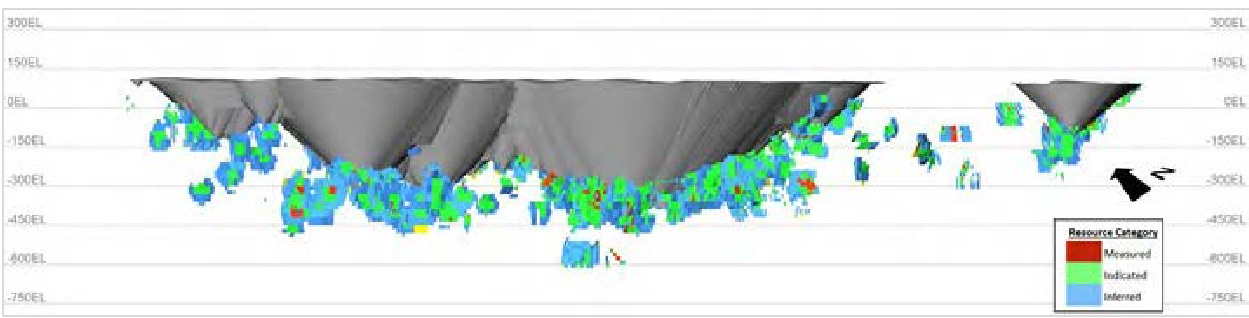


Figure 14-31: Underground Mineral Resource Estimate category long section looking due northeast

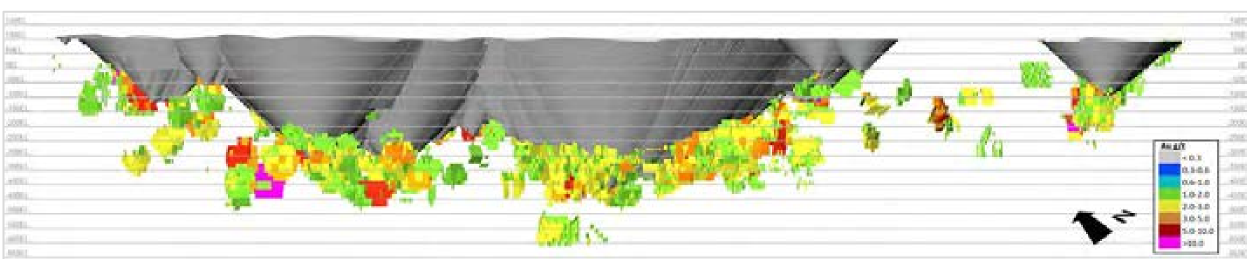


Figure 14-32: Main and NW Areas gold grade block model cross-sections

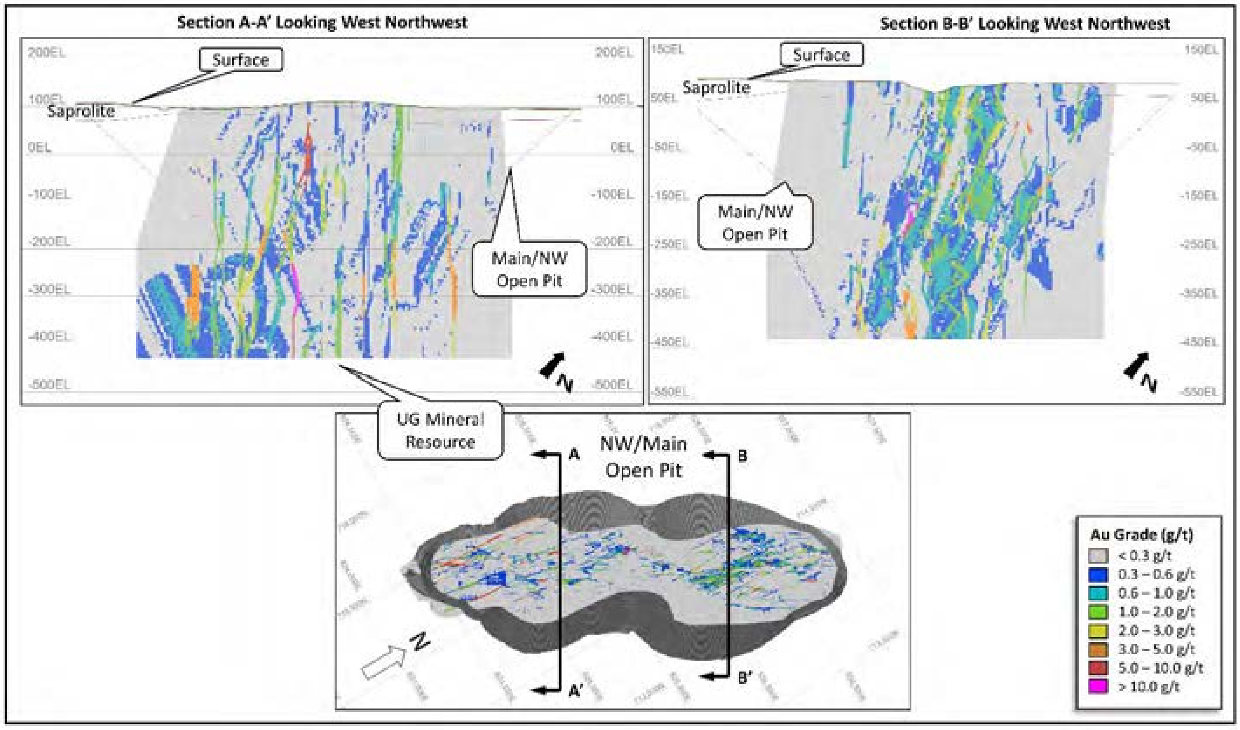


Figure 14-33: Toroparu Main and NW gold block model cross-section

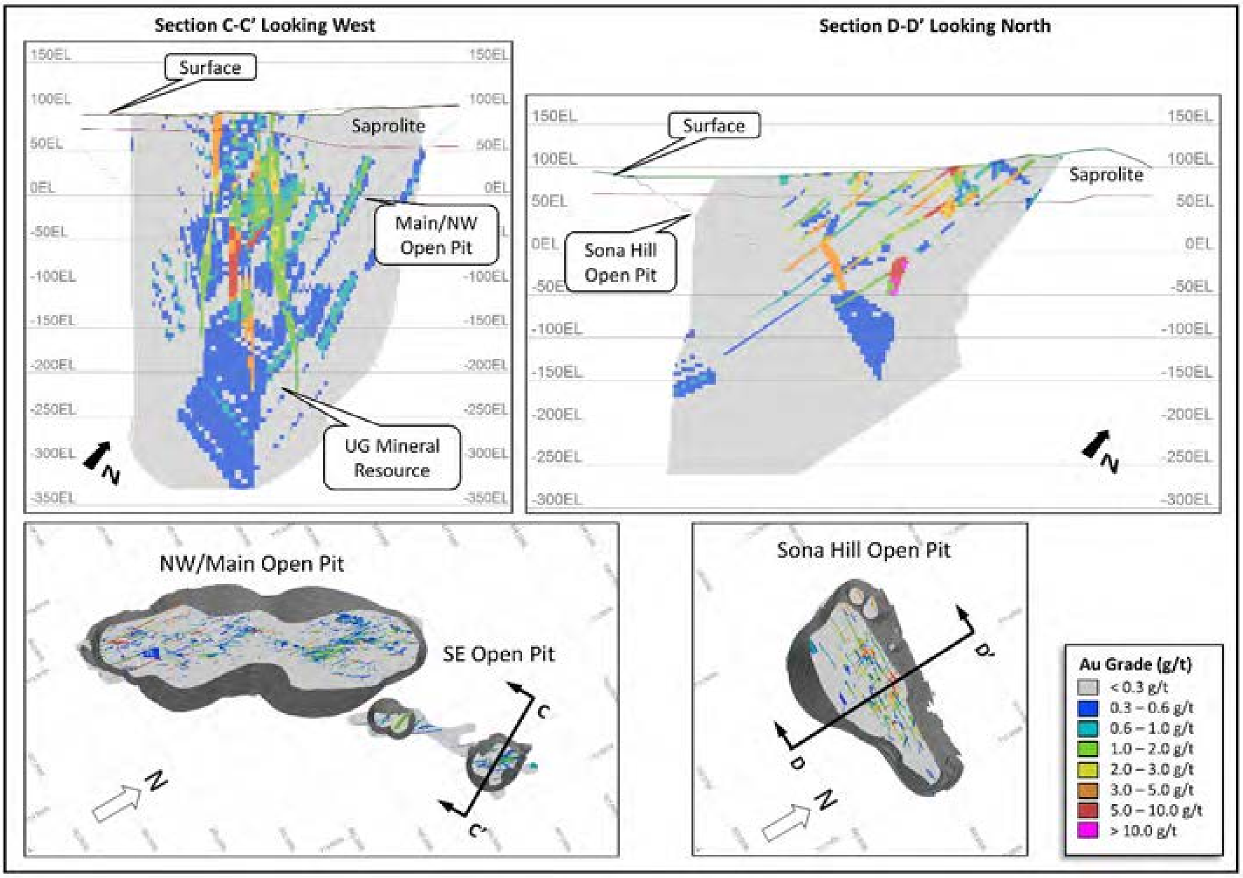


Figure 14-34: SE Area and Sona Hill Deposit gold grade block model cross-sections

14.11.2 Toroparu Deposit Main/NW and SE, and Sona Hill Open Pit Resource Estimate

Table 14-35 displays the open pit Mineral Resource Estimate classification of the Toroparu Main/NW and SE and Sona Hill Deposits summarized by saprolite and fresh rock types.

Table 14-35: Open Pit Resources by Rock Type

Area	Resource Category	Tonnes ('000s)	Au (g/t)	Ag (g/t)	Cu (%)	Au oz ('000s)	Ag oz ('000s)	Cu lb ('000s)
SAPROLITE								
Main/ NW	Measured	5,725	1.23	1.40	0.11	227	258	13,349
	Indicated	3,171	1.61	1.23	0.09	164	126	6,608
	Measured & Indicated	8,896	1.37	1.34	0.10	391	384	19,957
	Inferred	307	1.60	1.07	0.07	16	11	466
Sona Hill	Measured	1,597	1.67	1.14	0.008	86	58	39,987
	Indicated	772	1.40	0.94	0.008	35	23	15,959
	Measured & Indicated	2,369	1.58	1.07	0.008	120	82	55,947
	Inferred	260	1.00	0.83	0.006	8	7	4,778
SE	Measured	527	1.17	n/a	0.05	20	n/a	555
	Indicated	458	1.15	n/a	0.06	17	n/a	650
	Measured & Indicated	985	1.16	n/a	0.05	37	n/a	1,205
	Inferred	4	1.83	n/a	0.05	252	n/a	4
All Saprolite	**Measured**	**7,849**	**1.32**	**1.25**	**0.08**	**333**	**316**	**14,198**
	Indicated	**4,401**	**1.52**	**1.05**	**0.08**	**216**	**149**	**7,393**
	Measured & Indicated	**12,250**	**1.39**	**1.18**	**0.08**	**548**	**465**	**21,591**
	Inferred	**572**	**1.33**	**0.95**	**0.04**	**24**	**17**	**506**
FRESH ROCK								
Area	**Resource Category**	**Tonnes ('000s)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Au oz ('000s)**	**Ag oz ('000s)**	**Cu lb ('000s)**
Main/ NW	Measured	92,345	1.20	1.17	0.11	3,582	3,485	224,763
	Indicated	59,360	1.55	0.89	0.10	2,968	1,703	130,949
	Measured & Indicated	151,705	1.34	1.06	0.11	6,550	5,187	355,712
	Inferred	3,712	1.58	0.62	0.08	188	74	6,651
Sona Hill	Measured	5,361	1.90	1.05	0.008	328	180	947
	Indicated	3,408	1.72	0.83	0.008	188	91	565
	Measured & Indicated	8,769	1.83	0.96	0.008	516	272	1,512
	Inferred	1,105	1.35	0.47	0.006	48	17	143
SE	Measured	4,594	1.16	n/a	0.04	170	n/a	4,271
	Indicated	1,945	1.15	n/a	0.05	72	n/a	2,113
	Measured & Indicated	6,540	1.15	n/a	0.04	242	n/a	6,384
	Inferred	5	1.53	n/a	0.03	0	n/a	4
All Fresh	**Measured**	**102,301**	**1.24**	**1.11**	**0.10**	**4,080**	**3,665**	**229,980**
	Indicated	**64,713**	**1.55**	**0.86**	**0.09**	**3,228**	**1,794**	**133,626**
	Measured & Indicated	**167,014**	**1.36**	**1.01**	**0.10**	**7,308**	**5,459**	**363,607**
	Inferred	**4,821**	**1.52**	**0.58**	**0.06**	**236**	**91**	**6,798**
TOTAL OPEN PIT RESOURCE								
Area	**Resource Category**	**Tonnes ('000s)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Au oz ('000s)**	**Ag oz ('000s)**	**Cu lb ('000s)**
All	**Measured & Indicated**	**179,264**	**1.36**	**1.03**	**0.097**	**7,857**	**5,924**	**385,198**
	Inferred	**5,393**	**1.50**	**0.63**	**0.061**	**260**	**109**	**7,305**

Figure 14-35 provides a plan view of the grade distribution of the Main and NW Areas of the Toroparu Deposit and the Sona Hill Deposit within the open pit shell.

Figure 14-36 illustrates the resource classification associated within the open pit shell above the cut-off grade of the Main/NW Zone and SE Areas of the Toroparu Deposit and the Sona Hill Deposit.

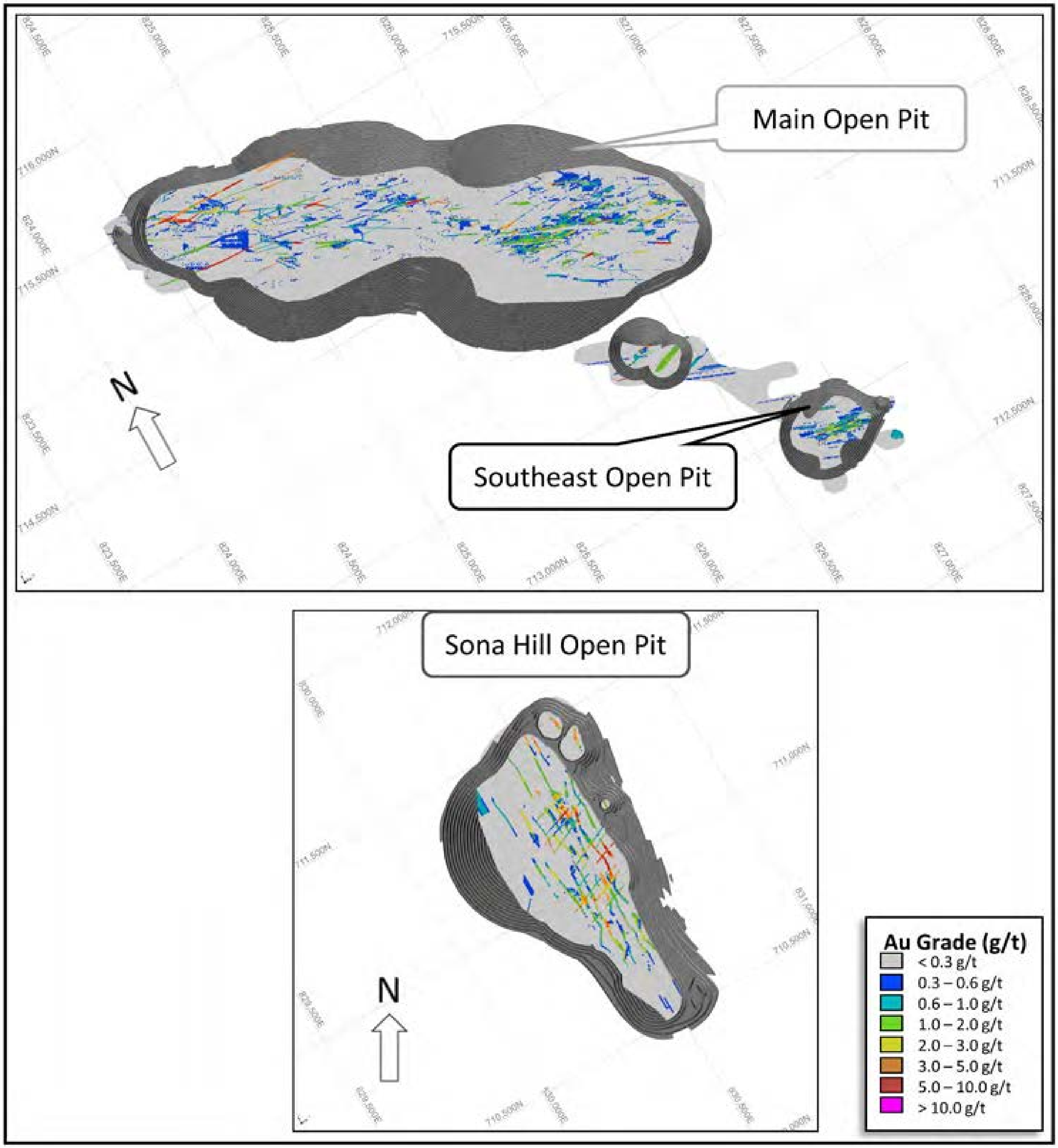


Figure 14-35: Open pit gold grade plan views – block model section at 90 m elevation (PSAD56-20 N)

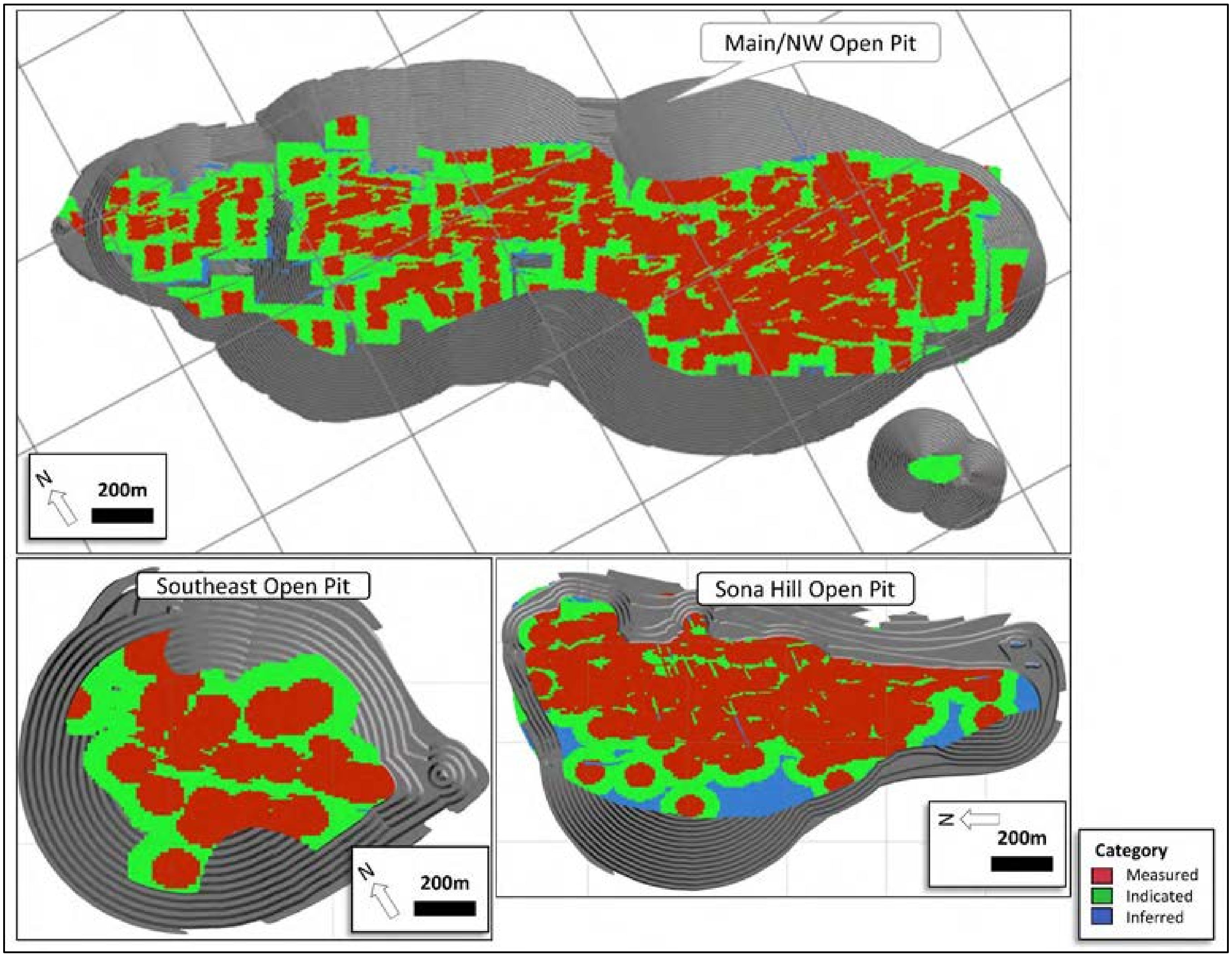


Figure 14-36: Open pit category views – block model section at 90 m elevation (PSAD56-20 N)

14.11.3 Cautionary Statement Regarding Mineral Resource Estimates

Until mineral deposits are actually mined and processed, Mineral Resources must be considered as estimates only. Mineral Resource Estimates that are not Mineral Reserves do not have demonstrated economic viability. The estimation of Mineral Resources is inherently uncertain, involves subjective judgment about many relevant factors and may be materially affected by, among other things, environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant risks, uncertainties, contingencies, and other factors described in the foregoing Cautionary Statements. The quantity and grade of reported "Inferred" Mineral Resource Estimates are uncertain in nature and there has been insufficient exploration to define "Inferred" Mineral Resource Estimates as an "Indicated" or "Measured" Mineral Resource and it is uncertain if further exploration will result in upgrading "Inferred" Mineral Resource Estimates to an "Indicated" or "Measured" Mineral Resource category. The accuracy of any Mineral Reserve and Mineral Resource Estimates is a function of the quantity and quality of available data, and of the assumptions made and judgments used in engineering and geological interpretation, which may prove to be unreliable and depend, to a certain extent, upon the analysis of drilling results and statistical inferences that may ultimately prove to be inaccurate. Mineral Reserve and Mineral Resource Estimates may have to be re-estimated based on, among other things: (i) fluctuations in mineral prices; (ii) results of drilling, and development; (iii) results of test stoping and other testing; (iv) metallurgical testing and other studies; (v) results of geological and structural modelling including stope design; (vi) proposed mining operations, including dilution; (vii) the evaluation of mine plans subsequent to the date of any estimates; and (viii) the possible failure to receive required permits, licences, and other approvals. It cannot be assumed that all or any part of a “Inferred”, “Indicated” or “Measured” Mineral Resource Estimate will ever be upgraded to a higher category. The Mineral Resource Estimates disclosed in this news release were reported using CIM Definition Standards for Mineral Resources and Mineral Reserves in accordance with National Instrument 43-101 of the Canadian Securities Administrators.

14.11.4 Mineral Resource Sensitivity to Reporting Cut-off

The sensitivity of the Mineral Resource Estimate to a range of cut-off grades for each category in the underground and open pit are contained below.

14.11.4.1 Underground

Underground Mineral Resource Estimate sensitivities can be found in Table 14-36.

Table 14-36: Toroparu Deposit Underground Resource Sensitivity

Resource Category	Tonnes (‘000s)	Au (g/t)	Ag (g/t)	Cu (%)	Au oz (‘000s)	Ag oz (‘000s)	Cu lb (‘000s)
Cut-Off = 1.5 g/t Au							
Measured	1,000	2.43	0.50	0.070	78	16	1,541
Indicated	6,799	2.77	0.41	0.088	606	90	13,152
Inferred	11,437	3.01	0.25	0.087	1,106	93	21,860
Cut-Off = 1.8 g/t Au							
Measured	**727**	**2.84**	**0.47**	**0.072**	**66**	**11**	**1,151**
Indicated	**4,978**	**3.21**	**0.41**	**0.091**	**514**	**66**	**9,93**
Inferred	**8,403**	**3.53**	**0.25**	**0.091**	**953**	**68**	**16,884**
Cut-Off = 2.0 g/t Au							
Measured	582	3.14	0.48	0.074	59	9	94
Indicated	4,007	3.55	0.41	0.094	457	52	8,3156
Inferred	6,803	3.93	0.24	0.096	860	51	14,458

14.11.4.2 Open Pit

Because the small open pit for the Toroparu Deposit has been fixed, sensitivities cannot be shown as they will overlap with underground resources. There is a 30 m pillar between the open pit and underground resources; thus, there is no overlap.

Open pit sensitivities for the Toroparu Deposit SE Area and Sona Hill Deposit can be found in Table 14-38.

Table 14-37 Toroparu Deposit Open Pit Sensitivity

Toroparu Deposit							
Resource Category	**Tonnes ('000s)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Au oz ('000s)**	**Ag oz ('000s)**	**Cu lb ('000s)**
Cut-off = 0.2 g/t							
Measured	180,034	0.81	0.96	0.09	4,682	5,537	345,626
Indicated	107,604	1.04	0.74	0.08	3,591	2,560	187,938
Inferred	5,041	1.31	0.60	0.07	212	97	8,005
Cut-off = 0.4 g/t							
Measured	**103,192**	**1.21**	**1.21**	**0.11**	**3,999**	**3,743**	**242,937**
Indicated	**64,934**	**1.54**	**1.54**	**0.10**	**3,221**	**1,828**	**140,320**
Inferred	**4,028**	**1.28**	**1.58**	**0.08**	**204**	**85**	**7,126**
Cut-off = 0.6 g/t							
Measured	69,673	1.68	1.28	0.12	3,757	2,878	177,444
Indicated	47,845	2.01	0.96	0.11	3,095	1,474	112,522
Inferred	3,880	1.89	0.65	0.07	236	81	6,349
Sona Hill Deposit							
Resource Category	**Tonnes ('000s)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Au oz ('000s)**	**Ag oz ('000s)**	**Cu lb ('000s)**
Cut-off = 0.2 g/t							
Measured	11,216	1.25	0.81	0.01	451	291	1,922
Indicated	6,053	1.23	0.68	0.01	240	133	987
Inferred	2,184	0.90	0.42	0.01	63	30	302
Cut-off = 0.4 g/t							
Measured	**6,958**	**1.85**	**1.07**	**0.01**	**413**	**239**	**1,241**
Indicated	**4,180**	**1.66**	**0.85**	**0.01**	**223**	**115**	**700**
Inferred	**1,365**	**1.28**	**0.54**	**0.01**	**56**	**24**	**179**
Cut-off = 0.6 g/t							
Measured	5,684	2.15	1.19	0.01	393	217	1,038
Indicated	3,362	1.95	0.96	0.01	210	103	576
Inferred	934	1.65	0.63	0.01	50	19	113

14.11.5 Comparison to the Previous Resource

A comparison of the Open pit only November 2021 Mineral Resource Estimate to the previous Open pot only 2018 Mineral Resource Estimate can be found in Table 14-38.

Table 14-38: Comparison to the Previous Resource

Location	September 2018					November 2021				
		Au Resources		Cu Resources			Au Resources		Cu Resources	
	Tonnes (000's)	Au (g/t)	Au (oz) (000's)	Cu (%)	Cu (MLbs)	Tonnes (000's)	Au (g/t)	Au (oz) (000's)	Cu (%)	Cu (MLbs)
Toroparu Deposit (Main/NW Areas)	**Measured and Indicated**					**Measured and Indicated**				
	153,052	1.00	4,918	0.10	336	184,969	1.42	8,437	0.10	396
Toroparu Deposit (Main/NW Areas)	**Inferred**					**Inferred**				
	29,698	0.81	774	0.04	27	13,795	2.74	1,213	0.08	24

14.12 Factors That May Affect the Mineral Resources

Areas of uncertainty that may materially impact the Mineral Resource Estimate include:

- Changes to long term metal price assumptions.
- Changes to the input values for mining, processing, and G&A costs to constrain the estimate.
- Changes to local interpretations of mineralization geometry and continuity of mineralized zones.
- Changes to the density values applied to the mineralized zones.
- Changes to metallurgical recovery assumptions.
- Changes in assumptions of marketability of the final product.
- Variations in geotechnical, hydrogeological, and mining assumptions.
- Changes to assumptions with an existing agreement or new agreements.
- Changes to environmental, permitting, and social licence assumptions.
- Logistics of securing and moving adequate services, labour, and supplies could be affected by epidemics, pandemics, and other public health crises, including COVID-19, or similar such viruses.

14.13 Comments on Section 14

The QP is not aware of any environmental, legal, title, taxation, socio-economic, marketing, political or other relevant factors that would materially affect the estimation of Mineral Resources that are not discussed in this Technical Report.

The QP is of the opinion that Mineral Resources were estimated using industry-accepted practices and conform to the 2014 CIM Definition Standards and 2019 CIM Best Practice Guidelines. Technical and economic parameters and assumptions applied to the Mineral Resource Estimate are based on Nordmin's internal calculations and feedback from the Company to determine if they were appropriate.

15 MINERAL RESERVE ESTIMATE

This section is not relevant to this Technical Report.

16 MINING METHODS

16.1 Introduction

The Project will be mined using open pit methods for the initial part of the extraction and in year 10 an underground mining method will be started. Both mining methods will continue to the end of the life of the asset.

16.2 Mineral Resources within the PEA Mine Plan

The mining methods used for open pit and underground are very well understood and used by other mining companies around the world. The mineral resources within the PEA mine plan estimate is effective as of December 1, 2021 and is presented in Table 16-1. The PEA models an open pit and an underground mine with mineral resources within the PEA mine plan containing 6.156 Moz of Au, 3.993 Moz of Ag and 240.2 Mlb of Cu (109.0 kt).

Measured, Indicated and Inferred resources were used for conversion to mineral resources within the PEA mine plan open pit and underground designs. The open pit mineral resources within the PEA mine plan are contained within the Toroparu Pit, Sona Hill Pit and SE Pit and are associated with 558 Mt of waste and a LoM stripping ratio of 5.99:1. The underground mineral resources within the PEA mine plan are contained below the Toroparu Pit.

The mineral resources within the PEA mine plan are valid at the time of estimation and include CoG assumptions made before the final PEA cash flow model was completed. SRK and Nordmin confirmed the overall project economics are favorable at the approximate four-year moving average Au price of US$1,500/oz Au, an average Ag price of US$20/oz Ag, and an average Cu price of US$3.13/lb Cu.

Table 16-1: Mineral Resources within the PEA Mine Plan

MINERAL RESOURCES WITHIN THE PEA MINE PLAN								
Area	Resource Category	Tonnes ('000s)	Au g/t	Ag g/t	Cu %	Contained Au Toz ('000s)	Contained Ag Toz ('000s)	Contained Cu Tonnes ('000s)
All Open Pits	Measured	60,117	1.41	1.36	0.11	2,728	2,633	64.6
	Indicated	31,407	1.74	1.12	0.09	1,756	1,126	29.8
	Measured & Indicated	91,525	1.53	1.28	0.10	4,499	3,769	94.5
	Inferred	1,593	1.62	0.89	0.07	83	45	1.1
	All Open Pits Subtotal	**93,118**	**1.53**	**1.27**	**0.10**	**4,567**	**3,804**	**95.5**
Underground	Measured	839	2.73	0.63	0.07	74	17	0.6
	Indicated	5,899	3.24	0.49	0.11	614	92	6.2
	Measured & Indicated	6,738	3.17	0.51	0.10	687	110	6.8
	Inferred	7,447	3.77	0.33	0.09	902	80	6.6
	Underground Subtotal	**14,185**	**3.48**	**0.41**	**0.09**	**1,589**	**189**	**13.4**
All Open Pits & Underground	Measured	60,956	1.43	1.35	0.11	2,802	2,650	65.3
	Indicated	37,306	1.98	1.02	0.10	2,369	1,219	36.0
	Measured & Indicated	98,262	1.64	1.23	0.10	5,187	3,878	101.3
	Inferred	9,040	3.39	0.43	0.09	985	125	7.7
	Grand Total	**107,302**	**1.78**	**1.16**	**0.10**	**6,156**	**3,993**	**109.0**

Source: SRK, 2021 & Nordmin, 2021

Open pit mineral resources within the PEA mine plan notes

- Open pit mineral resources within the PEA mine plan:
  - The open pit mineral resources within the PEA mine plan are based on a block by block net smelter return calculation based on an Au price of US$1,500/oz, Ag price of US$20.00/oz and Cu price of US$3.13/lb. The PEA cash flow base case used an Au price of US$1,500/oz., Ag price of US$20.20/oz and Cu price of US$3.13/lb;
  - The open pit mineral resources within the PEA mine plan assume complete mine recovery;
  - The open pit mineral resources within the PEA mine plan are diluted at approximately 15-30% (further to dilution inherent in the resource model and assumes selective mining unit of 5 m x 5 m x 5 m for Main and NW Pits and 2.5 m x 2.5 m x 5m for Sona Hill and SE pits);

- Contained in situ gold ounces do not include metallurgical ACO recoveries of 83.6% Cu and 80.2% Au and gold LCO recoveries of 92.2%;
    - Waste tonnes within the open pit is 558 Mt at a strip ratio of 5.99:1 (waste to ore);
    - Costs assumptions are: Mining Costs = US$2.30/t moved, Processing/Tailings Costs = US$15.50/t processed, G&A Costs = $5.95/t processed;
    - An open pit CoG of 0.5 g/t-Au saprolite and 0.5 g/t-Au fresh rock was applied to open pit resources constrained by the ultimate pit design; and
    - The open pit mineral resources within the PEA mine plan estimate for the Project was calculated by Fernando P. Rodrigues, BSc, MBA MMSAQP #01405QP of SRK Consulting, Inc. in accordance with the Canadian Securities Administrators National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and generally accepted Canadian Institute of Mining, Metallurgical and Petroleum "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines ("CIM Guidelines").
- Underground mineral resources within the PEA mine plan:
    - The underground mineral resources within the PEA mine plan were prepared by B. Wissent, BEng of Nordmin Engineering Ltd., in accordance with NI 43-101 and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. This estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues;
    - The underground mineral resources within the PEA mine plan are based on selected MSO wireframes generated at Au cut-off of 2.0 g/t based on an Au price of US$1,500/oz. A small amount of the underground mineral resources within the PEA mine plan is based on material from development with a marginal Au diluted cut-off of 1.25 g/t. The PEA cash flow base case used an Au price of US$1,500/oz., Ag price of US$20.20/oz and Cu price of US$3.13/lb;
    - The underground mineral resources within the PEA mine plan assumes mining recovery at approximately 80% to 92.5% for LHOS and 100% for development;
    - Contained in situ gold ounces do not include metallurgical recoveries;
    - Underground mineral resources within the PEA mine plan are diluted at approximately 12% for LHOS and 5% for development; and
    - Costs assumptions are: Mining Costs = US$36.00/t processed, Processing/Tailings Costs = US$15.50/t processed, G&A Costs = US$6.00/t processed, Operating Cost Marginal Allowance (10%) = US$5.80/t processed.
- Mineral resources within the PEA mine plan tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding;
- "g/t" = gram per metric tonne, "Toz" = troy ounces; and
- Mineral resources within the PEA mine plan effective date: December 1, 2021.

16.3 Open Pit Evaluation

Open pit mining will be by open pit methods using hydraulic excavators and wheel loaders loading articulated dump trucks for waste and economic material haulage. Mining activities at the Toroparu and Sona Hill mining operations will include removal of growth medium (topsoil), free-digging, drilling, blasting, loading, hauling and mining support activities.

16.3.1 Open Pit Mineral Resources within the PEA Mine Plan

16.3.1.1 Conversion Assumptions, Parameters and Methods

Conversion assumptions (e.g., model dilution, mining recovery, process recovery, cut-off grade calculation, pit optimization and costs) were taken into consideration to calculate the mineral resources within the PEA mine plan estimate.

The following steps were used to calculate the mineral resources within the PEA mine plan:

1. Apply mining dilution to resource block model (using 3-D techniques);
2. Estimate costs and process recoveries;
3. Input optimization parameters into pit optimizer to calculate nested pits using different Au selling prices (Measured, Indicated and Inferred resources were included as mineral resources within the PEA mine plan);
4. Select pit optimization shell based on strip ratio, revenue, grade distribution, discounted cash flow, cash costs, equipment selection sizes, pit footprint, depth of pit, minimum mining widths, cut-off grade, processing plant size and many other factors;
5. Complete detailed phase design with ramp access to all benches;
6. Develop multiple trade-off mine plans based on different processing rates (quarterly periods for the mine life);
7. Develop scoping study level truck haulage estimates;
8. Complete detailed mine cost estimates based on detailed mine plan;
9. Prepare a discounted cash flow based on all capital and operating cost inputs; and
10. Select a final mine plan and cash flow followed by reported mineral resources within the PEA mine plan.

16.3.1.2 Open Pit Model Grade Dilution

The mineralized deposit shell was developed by using two Au cut-offs (higher-grade and lower grade shells). SRK used the higher-grade Au shell for grade estimation while the lower grade g/t Au shell was used to calculate the dilution outside of the lower grade Au shell. SRK calculated the dilution using the following method and location:

- Main Pit and NW Pit areas: A SMU for these areas are assumed to be 5 x 5 x 5 m. Sublocked original models have a sublocked size of 0.5 x 0.5 x 2.5 m. A reblocked exercise to 5 x 5 x 5 m has been estimated. Main and NW areas dilution is estimated to approximately 20 to 25 % dilution.
- Sona Hill and SE Pit areas: A SMU for these areas are assumed to be 2.5 x 2.5 x 5 m. Sublocked original models have a sublocked size of 0.5 x 0.5 x 2.5 m. A reblocked exercise to 2.5 x 2.5 x 5 m has been estimated. Main and NW areas dilution is estimated at approximately 15% dilution.

16.3.1.3 Open Pit Potential Mineral Resources within the PEA Mine Plan Estimate

The estimate of open pit mineral resources within the PEA mine plan is effective as of December 1, 2021 and is presented in Table 16-2. The PEA models an open pit mine with mineral resources within the PEA mine plan containing 4.567 Moz of Au, 3.804 Moz of Ag and 210.6 Mlb of Cu (95.5 kt).

Measured, Indicated and Inferred resources were used for conversion to mineral resources within the PEA mine plan within the PEA ultimate pit designs. The mineral resources within the PEA mine plan (in-pit) are based on Au CoGs that vary depending on cyanide consumption. The average CoG is approximately 0.4 g/t Au.

The mineral resources within the PEA mine plan are contained within the Toroparu Pit, Sona Hill Pit and SE Pit and are associated with 558 Mt of waste and a LoM stripping ratio of 5.99:1.

The mineral resources within the PEA mine plan are valid at the time of estimation and include CoG assumptions made before the final PEA cash flow model was completed. SRK confirmed the overall project economics are favorable at the approximate four-year moving average Au price of US$1,500/oz Au, an average Ag price of US$20/oz Ag, and an average Cu price of US$3.13/lb Cu.

Table 16-2: Open Pit Mineral Resources within the PEA Mine Plan

OPEN PIT MINERAL RESOURCES WITHIN THE PEA MINE PLAN								
Area	**Resource Category**	**Tonnes (‘000s)**	**Au g/t**	**Ag g/t**	**Cu %**	**Contained Au Toz (‘000s)**	**Contained Ag Toz (‘000s)**	**Contained Cu Tonnes (‘000s)**
Main/NW	Measured	49,997	1.39	1.51	0.12	2,234	2,426	62.5
	Indicated	27,235	1.79	1.20	0.11	1,563	1,054	28.8
	Measured & Indicated	77,233	1.53	1.40	0.12	3,798	3,480	91.3
	Inferred	1,188	1.76	0.94	0.09	67	36	1.1
	Main/NW Subtotal	**78,420**	**1.53**	**1.39**	**0.12**	**3,865**	**3,515**	**92.4**
Sona Hill	Measured	6,455	1.63	1.00	0.01	338	207	0.5
	Indicated	2,612	1.53	0.86	0.01	129	72	0.2
	Measured & Indicated	9,067	1.60	0.96	0.01	482	289	0.8
	Inferred	397	1.21	0.76	0.01	15	10	0.0
	Sona Hill Subtotal	**9,465**	**1.58**	**0.95**	**0.01**	**482**	**289**	**0.8**
SE	Measured	3,665	1.32	-	0.04	156	-	1.6
	Indicated	1,560	1.27	-	0.05	64	-	0.8
	Measured & Indicated	5,225	1.31	-	0.05	219	-	2.4
	Inferred	8	1.54	-	0.04	0	-	0.0
	SE Subtotal	**5,233**	**1.31**	**-**	**0.05**	**220**	**-**	**2.4**
All Pits	Measured	60,117	1.41	1.36	0.11	2,728	2,633	64.6
	Indicated	31,407	1.74	1.12	0.09	1,756	1,126	29.8
	Measured & Indicated	91,525	1.53	1.28	0.10	4,499	3,769	94.5
	Inferred	1,593	1.62	0.89	0.07	83	45	1.1
	Grand Total	**93,118**	**1.53**	**1.27**	**0.10**	**4,567**	**3,804**	**95.6**

Source: SRK, 2021

16.3.1.4 Relevant Factors

There is no material mining, metallurgical, infrastructure, permitting and other factors that could affect the mineral resources within the PEA mine plan.

16.3.2 Parameters Relevant to Mine or Pit Designs and Plans

16.3.2.1 Pit Slope Geotechnical Design Criteria

16.3.2.1.1 Open Pit Geotechnical Site Investigation Programs

The PEA mining plan includes the Main, NW, SE and Sona Hill Pits. The Main, NW and SE pits are located within the Toroparu Deposit and the Sona Hill Pit is located within the Sona Hill deposit. The SE pit is located 1 km southeast of the Main and NW Pits and the Sona Hill Pit is located approximately 5 km southeast of the Main and NW Pits.

Knight Piésold Ltd. (KP) has been retained as a geotechnical consultant to provide geotechnical design criteria for open pit slopes at the Project. KP conducted a geotechnical site investigation program in 2010/2011 in support of the Main Pit slope design (KP Report, Ref. No. VA201-358/2-1, April 2013). KP followed up with a more detailed pit slope design in 2014 to include the SE Pit into the mining plan (KP Report, Ref. No. VA201-358/3-1, September 2014). KP completed a supplementary geotechnical site investigation program in 2018 to collect geotechnical data for the proposed Sona Hill Pit and to fill the data gaps of the SE Pit (KP Report Ref. No. VA201-358/4-1, October 2018).

The site investigation programs included visual inspections, oriented core geomechanical logging and sampling, field permeability testing, piezometer instrumentation and monitoring, downhole televiewer surveys, and laboratory rock and soil testing.

The collected geological, structural, rock mass, and hydrogeological data from three deposit sites were analyzed in conjunction with exploration drill hole logs, geology models, structural models, and available hydrogeological monitoring data. Simplified geotechnical models were developed for pit slope stability assessments.

16.3.2.1.2 Geotechnical Characterization

The stability of open pit rock slopes is typically controlled by wall geology, structural geology, rock mass characteristics and hydrogeological conditions.

The Project site is covered by a 30 to 40 m thick sequence of surficial saprolite. The bedrock geology of the Toroparu Deposit is dominated by a massive volcanic and metasedimentary assemblage. The fresh bedrock is comprised of massive volcanics, mixed facies, granodiorite and dykes, which have similar geomechanical properties. Two major geotechnical domains, Saprolite and Fresh Bedrock, were defined for pit slope geotechnical assessment for the Toroparu Deposit. The Fresh Bedrock domain were further divided into Intrusives and Cataclastic Hydrothermal Facies for the Sona Hill deposit due to some distinguishing structural features in the later sub domain.

Rock mass was characterized by using various geomechanical indices estimated from drill core, including intact rock Unconfined Compressive Strength (UCS), RQD, and Rock Mass Rating (RMR). The intact rock strengths were found to be STRONG to VERY STRONG. The RQD values were generally found to be high within all bedrock units. Combining the intact rock properties and characteristics of the observed rock fabric, the rock mass quality at the Toroparu Deposit is classified GOOD to VERY GOOD.

The Toroparu Deposit is interpreted to be between three major lineaments: the west-northwest oriented Puruni Shear, the Wynamu Fault and Majuba Hill Fault, both oriented north-northeast. A north-south trending, westerly dipping, low angle shear zone has been delineated within the entire Sona Hill deposit. Each deposit appears to have a predominant structural orientation in related to identified major structural features.

Groundwater levels were found to be near surface. Local scale groundwater flow systems will largely control the overall flow patterns, with recharge occurring on the relatively higher ground and discharge focused in adjacent localized topographic lows and the main rivers (Puruni River and Wynamu River). The hydraulic conductivity is relatively high adjacent to the saprolite/fresh bedrock contact and gradually decreases with depth. Three hydrogeological domains, Upper, Middle and Lower, were defined throughout the Toroparu Deposit, with hydraulic conductivities in the order of 10^{-5}, 10^{-6}, and 10^{-7} cm/s, respectively. Hydrogeological domains were not well defined for the Sona Hill Deposit due to limited testing data to date.

16.3.2.1.3 Slope Stability Analyses

Preliminary pit shells were utilized for various pit slope stability analyses. A total of four design sectors (M-North, M-East, M-South, and M-West) were defined for the Main/ NW Pits, based on the orientations of pit walls and structural features. The SE Pit was divided into three design sectors, namely the SE-Northwest, SE-Northeast, and SE-Southeast Sectors. The Sona Hill Pit was divided into four sectors, namely the SH-Northeast, SH-Southeast, SH-Southwest, and SH-Northwest Sectors. Sub sectors were also delineated to differentiate the Saprolite and Fresh Bedrock domains in each sector.

Rock mass structural features measured from oriented drill core and televiewer surveys were used in kinematic analyses to identify possible structurally controlled failure modes within rock slopes. Adverse structural features were identified mainly in the M-North, SE-Northeast, SE-Southeast, SH-Northeast, and SH-Southeast Sectors. Bench geometries were selected to reduce the potential planar, wedge sliding, and toppling which can affect bench face integrity and reduce their effectiveness.

Saprolite and rock mass slope stability analyses were performed to estimate the Factor of Safety (FoS) against large scale, multiple bench failures through saprolite soils and rock mass. The stability of the Saprolite slopes is dictated by material strength and porewater pressure conditions. Given the nature of competent rock mass, the risk of large scale circular failure in bedrock is low.

16.3.2.1.4 Recommended Pit Slope Configurations

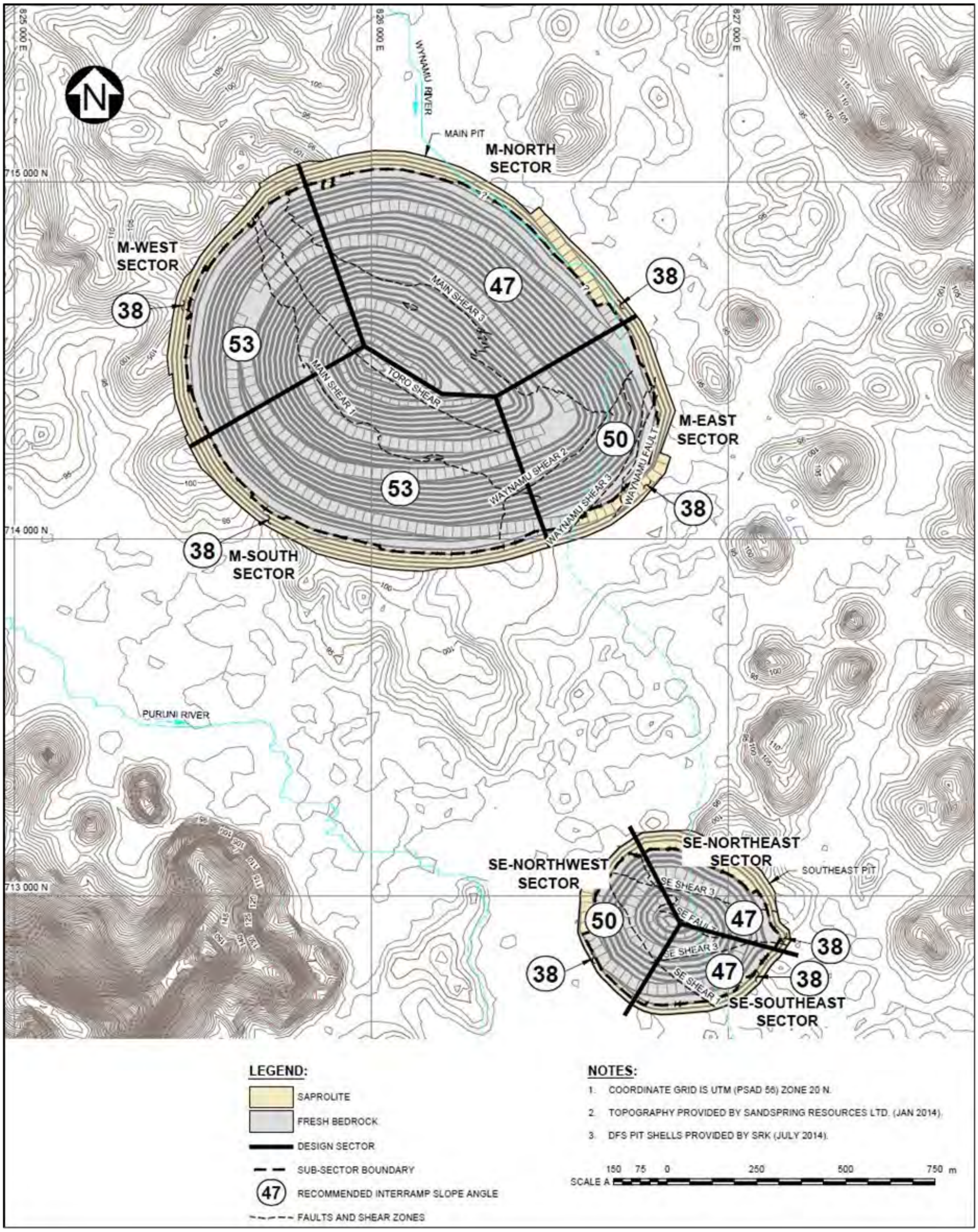
The overall objective of the pit slope design was to determine the steepest practical slope angles to maximize extraction of the mineral resources. The pit slope design was based on the available geotechnical database, geological/structural models and corresponding stability analysis results. This work led to the development of pit slope design parameters for benches, inter-ramp slopes, and overall slopes in each of the pit design sectors of the proposed open pits. A 10-m high single bench was selected for open pit slope geotechnical assessment.

A bench face angle of 65° is expected to be appropriate for the Saprolite slopes provided that adequate catch benches are emplaced. A 10 m high single bench configuration with a minimum bench width of 8 m is recommended for the Saprolite slopes, assuming moderately sized mining equipment being used for pit development.

In Fresh Bedrock, a slightly flatter bench face angle of 65° is deemed to be appropriate for the M-North, SE-Northeast, and SE-Southeast Sectors due to potential planar daylighting and/or minor wedge/toppling. A bench face angle of 70° is achievable for the M-East, and SE-Northwest, SH-Northeast, and SH-Southeast Sectors despite the minor potential for planar/wedge sliding and potential toppling caused by inferred faults. A steeper bench face angle of 75° can be applied for the M-South, M-West, SH-Southwest and SH-Northwest Sectors as foreseeable kinematic control is absent. Given the nature of competent rock mass, 20 m high double benching configurations can be considered for the pit walls developed within the Fresh Bedrock. The bench width is recommended to be between 9.5 and 10 m to catch possible ravelling and rock fall debris.

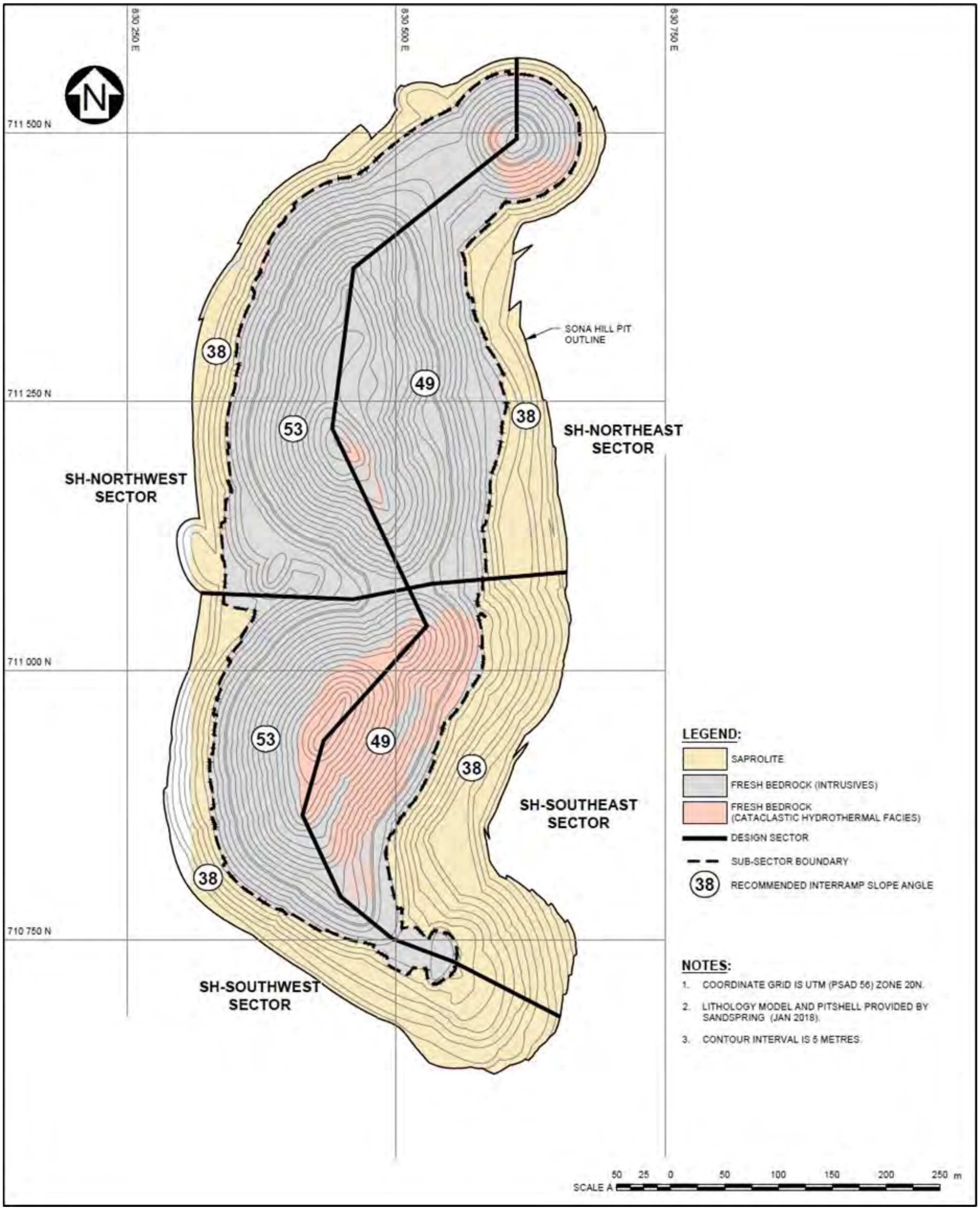
The inter-ramp slope angles are typically determined by the bench geometry. A shallow inter-ramp angle of 38° is recommended in the Saprolite slopes. The bedrock slopes in the M-North, SE-Northeast, and SH-Southeast Sectors are limited to an inter-ramp angle of 47° due to the presence of adverse planar features. A 50° inter-ramp angle is recommended for the M-East and SE-Northwest Sectors where the potential for minor adverse structural features are identified. A slightly flatter 49° inter-ramp angle is recommended for the SH-Northeast and SH-Southeast Sectors where a low angle shear zone/foliation feature is present but is not expected to have major adverse impact to the pit walls. A steeper inter-ramp slope angle of 53° can be applied for the slopes with fewer kinematic controls, including the M-South, M-West, SH-Southwest, and SH-Northwest Sectors.

Recommended inter-ramp slope angles for the Main/NW and Southeast Pits and for the Sona Hill Pit are illustrated on Figure 16-1 and Figure 16-2, respectively. It should be noted that the pit shells presented in the design sector figures are the 2018 study models and not necessarily consistent with the current PEA pit models.



Source: KP, 2018

Figure 16-1: Recommended pit slope angles (Main/NW and SE Pits)



Source: KP, 2018

Figure 16-2: Recommended pit slope angles (Sona Hill Pit)

Recommended pit slope configurations for the proposed final pits are summarized Table 16-3.

Table 16-3: Recommended Pit Slope Configurations for All Pits

Pit	Pit Design Sector		Wall Dip Direction (°)	Bench Face Angle (°)	Bench Height (m)	Bench Width (m)	Inter-ramp Angle (°)
All	Saprolite			65	10	8	38
Main/NW	Fresh Bedrock	M-North	160 to 240	65	20	9.5	47
		M-East	240 to 340	70	20	9.5	50
		M-South	340 to 360 000 to 060	75	20	10	53
		M-West	060 to 160	75	20	10	53
Southeast		SE-Northeast	160 to 250	65	20	9.5	47
		SE-Southeast	250 to 360 360 to 030	65	20	9.5	47
		SE-Northwest	030 to 160	70	20	9.5	50
Sona Hill		SH-Northeast	180 to 270	70	20	10	49
		SH-Southeast	270 to 315	70	20	10	49
		SH-Southwest	315 to 360 000 to 090	75	20	10	53
		SH-Northwest	090 to 180	75	20	10	53

Source: KP, 2018

A 20 m wide catch bench should be placed immediately below the saprolite/bedrock contact to intersect the surface run-off and seepage inflow and to provide additional containment capacity for potential saprolite ravelling during wet seasons. It is also recommended that the maximum height of inter-ramp slope in the Fresh Bedrock domain be limited to 200 m in the Main/NW Pit. The overall slope angles are expected to be 5° to 8° flatter than the inter-ramp slopes in bedrock after the flatter Saprolite slopes, wider catch bench, and spiral haulage ramps are incorporated.

This design has a number of operational constraints including low damage wall blasting, regular pit wall scaling and debris cleanout, effective pit dewatering and slope depressurization, and commitments of piezometer instrumentation and slope monitoring for the critical slopes.

16.3.2.2 Hydrogeological and Pit Dewatering Design Criteria

16.3.2.2.1 Pit Hydrogeological Site Investigations

Hydrogeological testing, instrumentation, and monitoring were integrated into the 2010 and 2018 open pit geotechnical site investigation programs to collect hydrogeological data in support of pit dewatering design. Hydrogeological data collected from the surrounding environmental monitoring wells were also utilized for hydrogeological characterization. Three hydrogeological domains, Upper, Middle and Lower, were defined throughout the Toroparu Deposit. Hydrogeological domains for the Sona Hill deposit have not been well defined.

16.3.2.2.2 Pit Inflow Estimate

Pit inflows will likely be dominated by precipitation at the Project site, as well as groundwater seepage from relatively fractured saprolite/bedrock transition zones and geological structures. The estimated base case groundwater inflows for the ultimate Main/NW and Southeast Pits are 23 L/s and 10 L/s, respectively. The average groundwater mine inflow to the proposed Sona Hill Pit at the end of mine life is estimated to be 15 L/s.

16.3.2.2.3 Pit Dewatering

The pit water management systems include perimeter dykes and diversion ditches, in-pit water collection ditches, and in-pit pumps and collection systems to transfer water from the open pits to discharge points.

KP developed a conceptual pit dewatering plan targeting the Main/NW Pit Saprolite/Bedrock contact zone, the Main/NW Pit deep zone, and the SE Pit. The pit dewatering systems have been designed to meet the combined requirements of run-off from mean annual precipitation, groundwater seepage inflow, and 1 in 100-year 24-hour storm event during mine operations. The in-pit pumps were sized to remove ponded stormwater within 7 days. The yearly based design flows for each dewatering system are summarized in Table 16-4, however, this preliminary estimate was based on the 2018 pit models with a shorter LoM schedule of 15 years than the final PEA mine production schedule of 22 years.

Table 16-4: Summary of Pit Design Pumping Flows (based on shorter 15-year LoM)

Year	Main/NW Pits				Southeast Pit		
	Surface Area ($x10^3$ m^2)	Pit Depth (m)	Contact Zone Pumping Rate (L/s)	Deep Zone Pumping Rate (L/s)	Surface Area ($x10^3$ m^2)	Pit Depth (m)	Pumping Rate (L/s)
-2	129	35	24	68			
-1	221	75	33	109	131	70	77
1	369	115	46	176	132	140	79
2	369	195	46	178	258	140	145
3	726	205	78	334	258	200	145
4	893	205	93	407			
5	897	235	93	412			
6	1,287	275	124	586			
7	1,287	315	124	586			
8	1,287	365	124	587			
9	1,287	365	124	587			
10	1,287	365	124	587			
11	1,287	365	124	587			
12	1,287	375	124	587			
13	1,287	405	124	588			
14	1,287	465	124	588			
15	1,287	525	124	588			

Source: KP, 2018

Pit dewatering concepts for the Sona Hill Pit were discussed but a conceptual dewatering plan was not developed due to the lack of key information (primarily topography data for the surrounding area).

16.3.2.3 Mine or Pit Optimization

16.3.2.3.1 Mineral Resource Models

Mineral resource block models were imported into Whittle™ and verified against the original mineral resource block model (block model), created in Vulcan™. The Vulcan™ block models subsequently were coded in preparation for optimization. This included diluting the block models to account for mining practices. The verification process indicated no material changes to the block model tonnages and grade during the process of importing into Whittle™. Table 16-5 shows the block model sizes used in the pit optimization.

Table 16-5: Block Model Block Sizes SMU

Item	Main/NW	Southeast	Sona Hill
X (m)	5	2.5	2.5
Y (m)	5	2.5	2.5
Z (m)	5	5	5

Source: SRK, 2021

16.3.2.3.2 Topographic Data

The most recent fully validated topographic data was used during construction of the block model. Sona Hill topography is based on the drill hole collar information and contoured manual surveying. The Main/NW and SE Pit areas topography is based on detailed LIDAR survey.

16.3.2.3.3 Optimization Constraints

The optimization process was restricted to classifications of Measured, Indicated and Inferred in accordance with the Canadian National Instrument guidelines for NI 43-101. For the purpose of the optimization, there were no production or processing limits used within Whittle™, and all material not classified as Measured or Indicated or Inferred was treated for calculation purposes as waste.

16.3.2.3.4 Optimization Parameters

The pit optimizations have been carried out using Whittle™ optimization software (Whittle™ Version 4.4). Metal prices, operating costs, process recoveries and other factors as described in this section were inputs to the Whittle™ software.

16.3.2.3.5 Mining Dilution

The block model as imported into Whittle™ was diluted through a 3D dilution study. The optimization process included factors of 0% mining dilution and 100% economic material recovery (as this was pre-coded into the block model). These parameters were historical but considered by SRK to be reasonable.

16.3.2.3.6 Discount Rate

The pit optimization process did not utilize a discounting factor. Inflation was not factored into the costs, which represent an indication of the "Current Prices" in the analysis.

The Lerchs-Grossmann algorithm (on which the Whittle™ software is based) produces a series of mathematically optimum pit shells directly linked to the Revenue Factor utilized if the maximum undiscounted cash flow is the selection criterion for optimization.

16.3.2.3.7 Geotechnical Parameters

For the Toroparu optimization, three geotechnical domains were utilized. For the upper Saprolitic zones, an overall wall angle of 28° was used. Below the saprolite, in the North-East corner of the proposed pit, an overall wall angle of 38° was used. For all other areas of the pit, an overall wall angle of 45° was used. These parameters are much shallower than the maximum recommended inter-ramp angles to account for ramp systems within the pit optimization runs.

For the SE Pit optimization, two geotechnical zones were utilized. For the upper Saprolitic zone, an overall wall angle of 28° was used. Below the saprolite, an overall wall angle of 40° was used. The overall wall angle includes any allowances for ramps within each wall.

16.3.2.3.8 Royalties

Royalties have been provided by the Company. Royalties of 8% for Au sales and 1.5% for Cu sales have been applied.

16.3.2.3.9 Mining Costs

SRK reviewed the proposed costs and modified the input values based on prior experience with similar projects. SRK has not applied an incremental cost to account for the increased cost of mining at depth.

Material has been classified either as saprolite or fresh rock, and a unique cost per tonne has been applied for each material type. For saprolite, the cost per tonne is US$1.90/t and for Fresh (or Sulphide) Rock, the cost per tonne is US$2.30/t. These values are different than the final calculated operating cost estimate, however they are reasonable estimates. Since the optimization phase of the Project, costs have been re-estimated and the differences in the final results are not material.

16.3.2.3.10 Processing Costs and Recoveries

The estimated processing costs for both deposits were supplied by Metifex and verified by SRK. Three processing methods have been identified with unique costs and Metifex compiled an average cost depending on the composition of the feed to the plant. The higher of these costs was considered for all material types for the optimization and it was assumed as US$15.50/t milled.

16.3.2.3.11 Other Costs

Due to the different processing methods, a series of other costs also require inclusion in the pit optimization. Table 16-6 summarizes the optimization parameters used. The total Combined PCOST and G7A is US$21.45/t processed.

Table 16-6: Optimization Parameters (Base Case)

Parameter	Unit	Value
Mining Dilution	%	15% to 30%
Mining Dilution Grade		0
Mining Recovery	%	100
Slope Angle	(°)	Variable
Mining Cost	US$/t	2.30
Mining Rate	Mtpa	42
Processing Rate	Mtpa	2.5
Process Recovery	%	Variable
Processing Costs	US$/t ore	15.50
General and Administration	US$/t ore	5.75
Sustaining Capital Cost	US$/t ore	0.20
Au Price	US$/oz	1,630
Ag Price	US$/oz	20.00
Cu Price	US$/lb	3.13
Au Royalty	% of Sales	8%
Ag Royalty	% of Sales	8%
Copper Royalty	% of Sales	1.5%

Parameter	Unit	Value
Doré Au NSR Deductions / Losses	% of Recovered	0.05%
Doré NSR Transport and Insurance	US$/oz	2.45
Doré NSR Refining Charges	US$/oz	0.48
Cu Concentrate Au NSR Deductions / Losses	% of Au Sales	5%
Cu Concentrate Au NSR Smelting and Refining	US$/oz	4.5
Cu Concentrate Ag NSR Deductions / Losses	% of Ag Sales	10%
Cu Concentrate Ag NSR Smelting and Refining	US$/oz	1.75
Cu Concentrate Cu NSR Deductions / Losses	% of Cu Sales	4.76%
Cu Concentrate Cu NSR Treatment	US$/t	92
Cu Concentrate Cu NSR Refining	US$/lb	0.092
Cu Concentrate Au/Ag/Cu NSR Insurance	% of Sales	0.167%
Cu Concentrate Freight and Marketing	US$/lb	149.08

Source: SRK, 2021

16.3.2.4 Optimization Process

To optimize both deposits, a series of nested pit shells were calculated over a range of RF. Each of the nested pit shells were generated based on the maximum undiscounted cash flow calculated for the applicable RF. The generated nested pit shells increase in size as the RF and maximum undiscounted cash flow also increase.

To determine the optimum pit shell and for reporting purposes within Whittle™, the reported cash flow for RF=1 has been used (corresponding to an Au price of US$1,630/oz).

As part of the optimization process, Whittle™ uses the pit tonnages from nested pits and calculates the cashflow based on RF=1. Therefore, nested pit shells generated for a RF less than 1 will have cash flows greater than those used to determine the physical nested pit shell. Nested pit shells generated at a RF greater than 1 will have cash flows less (even negative) than those used to determine the physical nested pit shell. This is because material is mined (in the larger pits) that is economic when the original RF is applied; however, when RF's greater than 1 are used, some material within the pit becomes uneconomic, thus reducing the cashflow of that pit shell.

16.3.3 Optimization Results

Table 16-7 tabulates the summarized results of the optimization process for the Main/NW Pits. These pit shells were used in mine planning to guide the designs of the pit phases.

Table 16-8 tabulates the summarized results of the optimization process for the SE Pit.

Table 16-9 tabulates the summarized results of the optimization process for the Sona Hill Pit. Material processed from Sona Hill will not be sent to the flotation circuit as it does not contain any Cu mineralization. For the PEA it has been assumed no Ag will be recovered from material processed from Sona Hill. However, some Ag will likely be recovered along with Au, in the Au recovery process.

Table 16-7: Pit Optimization Results for Main and NW Pits Area

Pit	Rev Factor	Au Sell Price	Ag Sell Price	Cu Sell Price	SR	Total Tonnes ('000s)	Ore Tonnes ('000s)	Waste Tonnes ('000s)	Process SAP CIP ('000s)	Process Fresh CIP ('000s)	Process Fresh FLOAT ('000s)	Au Contained Toz ('000s)	Cu Contained Lbs ('000s)	Ag Contained Toz ('000s)	Au Recovered Toz ('000s)	Cu Recovered Lbs ('000s)	Ag Recovered Toz ('000s)	Au g/t	CU%	AG g/t
1	0.30	$ 489	$ 6.00	$ 0.94	4.59	30,303	5,418	24,885	2,378	2,939	101	720	518	149	690	456	124	4.13	0.00	0.85
2	0.35	$ 571	$ 7.00	$ 1.10	4.11	39,303	7,685	31,618	3,524	3,691	470	855	3,309	284	818	2,912	233	3.46	0.02	1.15
3	0.40	$ 652	$ 8.00	$ 1.25	3.81	50,847	10,573	40,274	4,579	5,028	966	1,010	6,162	414	963	5,422	335	2.97	0.03	1.22
4	0.45	$ 734	$ 9.00	$ 1.41	3.85	76,859	15,860	61,000	5,439	7,810	2,610	1,281	15,918	726	1,214	14,007	571	2.51	0.05	1.42
5	0.50	$ 815	$ 10.00	$ 1.57	4.49	176,002	32,047	143,955	6,272	18,925	6,850	2,109	37,583	1,527	1,983	33,073	1,171	2.05	0.05	1.48
6	0.55	$ 897	$ 11.00	$ 1.72	5.60	329,040	49,874	279,165	6,722	31,779	11,374	3,058	62,874	2,339	2,865	55,329	1,776	1.91	0.06	1.46
7	0.60	$ 978	$ 12.00	$ 1.88	5.89	401,981	58,376	343,605	6,857	38,045	13,474	3,476	84,816	2,655	3,251	74,638	2,013	1.85	0.07	1.41
8	0.65	$ 1,060	$ 13.00	$ 2.03	5.72	515,613	76,725	438,888	6,947	53,824	15,954	4,191	97,945	3,407	3,921	86,192	2,579	1.70	0.06	1.38
9	0.70	$ 1,141	$ 14.00	$ 2.19	6.65	925,813	121,003	804,810	7,068	91,669	22,265	6,156	158,430	4,492	5,756	139,419	3,394	1.58	0.06	1.15
10	0.75	$ 1,223	$ 15.00	$ 2.35	6.89	1,016,122	128,825	887,297	7,076	98,869	22,880	6,518	162,205	4,677	6,097	142,740	3,534	1.57	0.06	1.13
11	0.80	$ 1,304	$ 16.00	$ 2.50	8.28	1,500,641	161,682	1,338,959	7,085	128,424	26,173	8,139	184,543	5,332	7,623	162,398	4,028	1.57	0.05	1.03
12	0.85	$ 1,386	$ 17.00	$ 2.66	8.33	1,544,124	165,455	1,378,669	7,087	132,034	26,334	8,292	185,344	5,389	7,768	163,103	4,071	1.56	0.05	1.01
13	0.90	$ 1,467	$ 18.00	$ 2.82	8.33	1,566,061	167,780	1,398,282	7,087	134,258	26,434	8,372	185,844	5,416	7,844	163,543	4,092	1.55	0.05	1.00
14	0.95	$ 1,549	$ 19.00	$ 2.97	8.49	1,649,761	173,922	1,475,839	7,090	140,140	26,691	8,617	186,664	5,456	8,076	164,264	4,122	1.54	0.05	0.98
15	**1.00**	**$ 1,630**	**$ 20.00**	**$ 3.13**	**9.02**	1,879,894	187,601	1,692,293	7,091	152,893	27,617	9,204	190,279	5,572	8,629	167,446	4,209	**1.53**	**0.05**	**0.92**
16	1.05	$ 1,712	$ 21.00	$ 3.29	9.08	1,908,367	189,398	1,718,969	7,092	154,577	27,729	9,272	191,400	5,586	8,694	168,432	4,220	1.52	0.05	0.92
17	1.10	$ 1,793	$ 22.00	$ 3.44	9.19	1,965,065	192,927	1,772,137	7,092	157,927	27,908	9,404	192,200	5,625	8,819	169,136	4,249	1.52	0.05	0.91
18	1.15	$ 1,875	$ 23.00	$ 3.60	9.24	1,993,422	194,722	1,798,700	7,093	159,689	27,940	9,467	192,384	5,633	8,879	169,298	4,256	1.51	0.04	0.90
19	1.20	$ 1,956	$ 24.00	$ 3.76	9.33	2,029,781	196,522	1,833,259	7,093	161,399	28,030	9,538	192,978	5,645	8,946	169,820	4,265	1.51	0.04	0.89

Table 16-8: Pit Optimization Results for SE Pit Area

Pit	Rev Factor	Au Sell Price	Ag Sell Price	Cu Sell Price	SR	Total Tonnes ('000s)	Ore Tonnes ('000s)	Waste Tonnes ('000s)	Process SAP CIP ('000s)	Process Fresh CIP ('000s)	Process Fresh FLOAT ('000s)	Au Contained Toz ('000s)	Cu Contained Lbs ('000s)	Ag Contained Toz ('000s)	Au Recovered Toz ('000s)	Cu Recovered Lbs ('000s)	Ag Recovered Toz ('000s)	Au g/t	CU%	AG g/t
1	0.30	$ 489	$ 6.00	$ 0.94	1.77	464	167	296	149	17	1	16	6	-	15	5	-	2.94	0.00	-
2	0.35	$ 571	$ 7.00	$ 1.10	1.81	785	280	505	231	44	5	22	16	-	21	14	-	2.44	0.00	-
3	0.40	$ 652	$ 8.00	$ 1.25	1.64	1,450	550	901	352	191	6	34	19	-	33	17	-	1.95	0.00	-
4	0.45	$ 734	$ 9.00	$ 1.41	1.73	4,912	1,802	3,110	507	1,255	40	89	131	-	85	116	-	1.53	0.00	-
5	0.50	$ 815	$ 10.00	$ 1.57	2.46	12,074	3,494	8,579	598	2,777	119	160	494	-	153	435	-	1.43	0.01	-
6	0.55	$ 897	$ 11.00	$ 1.72	2.62	13,706	3,790	9,915	650	3,004	137	174	576	-	165	507	-	1.42	0.01	-
7	0.60	$ 978	$ 12.00	$ 1.88	2.64	14,447	3,974	10,473	664	3,172	138	180	578	-	172	509	-	1.41	0.01	-
8	0.65	$ 1,060	$ 13.00	$ 2.03	2.93	18,191	4,627	13,563	694	3,756	178	205	762	-	195	671	-	1.38	0.01	-
9	0.70	$ 1,141	$ 14.00	$ 2.19	3.32	23,327	5,396	17,931	699	4,493	204	234	856	-	223	753	-	1.35	0.01	-
10	0.75	$ 1,223	$ 15.00	$ 2.35	3.54	26,562	5,845	20,717	708	4,924	213	251	878	-	239	773	-	1.33	0.01	-
11	0.80	$ 1,304	$ 16.00	$ 2.50	3.70	28,713	6,109	22,604	718	5,165	226	260	912	-	248	802	-	1.33	0.01	-
12	0.85	$ 1,386	$ 17.00	$ 2.66	3.82	30,334	6,292	24,042	721	5,343	228	267	917	-	254	807	-	1.32	0.01	-
13	0.90	$ 1,467	$ 18.00	$ 2.82	3.93	31,606	6,408	25,198	721	5,449	239	271	945	-	258	831	-	1.32	0.01	-
14	0.95	$ 1,549	$ 19.00	$ 2.97	3.96	31,911	6,439	25,472	721	5,478	241	272	950	-	259	836	-	1.31	0.01	-
15	**1.00**	**$ 1,630**	**$ 20.00**	**$ 3.13**	**4.26**	35,230	6,691	28,538	721	5,716	254	281	987	-	268	868	-	**1.31**	**0.01**	**-**
16	1.05	$ 1,712	$ 21.00	$ 3.29	4.29	35,502	6,709	28,793	722	5,733	254	282	987	-	268	868	-	1.31	0.01	-
17	1.10	$ 1,793	$ 22.00	$ 3.44	5.12	43,272	7,076	36,196	730	6,065	281	299	1,055	-	284	928	-	1.31	0.01	-
18	1.15	$ 1,875	$ 23.00	$ 3.60	5.31	44,975	7,126	37,849	730	6,111	285	302	1,065	-	287	937	-	1.32	0.01	-
19	1.20	$ 1,956	$ 24.00	$ 3.76	5.59	48,036	7,286	40,750	730	6,265	290	308	1,084	-	293	954	-	1.31	0.01	-

Table 16-9: Pit Optimization Results for Sona Hill Pit Area

Pit	Rev Factor	Au Sell Price	Ag Sell Price	Cu Sell Price	SR	Total Tonnes ('000s)	Ore Tonnes ('000s)	Waste Tonnes ('000s)	Process SAP CIP ('000s)	Process Fresh CIP ('000s)	Process Fresh FLOAT ('000s)	Au Contained Toz ('000s)	Cu Contained Lbs ('000s)	Ag Contained Toz ('000s)	Au Recovered Toz ('000s)	Cu Recovered Lbs ('000s)	Ag Recovered Toz ('000s)	Au g/t	CU%	AG g/t
1	0.30	$ 489	$ 6.00	$ 0.94	1.17	2,609	1,201	1,407	613	588	-	99	-	50	89	-	43	2.57	-	1.29
2	0.35	$ 571	$ 7.00	$ 1.10	1.62	7,141	2,728	4,413	1,534	1,194	-	188	-	115	168	-	100	2.14	-	1.32
3	0.40	$ 652	$ 8.00	$ 1.25	1.69	8,606	3,202	5,404	1,667	1,536	-	214	-	133	191	-	115	2.08	-	1.29
4	0.45	$ 734	$ 9.00	$ 1.41	1.87	11,675	4,068	7,607	1,799	2,269	-	260	-	159	230	-	137	1.98	-	1.22
5	0.50	$ 815	$ 10.00	$ 1.57	2.31	17,865	5,393	12,472	1,910	3,483	-	326	-	197	286	-	170	1.88	-	1.14
6	0.55	$ 897	$ 11.00	$ 1.72	2.47	21,492	6,186	15,306	1,989	4,197	-	363	-	218	316	-	188	1.83	-	1.09
7	0.60	$ 978	$ 12.00	$ 1.88	2.59	24,099	6,713	17,386	2,082	4,631	-	386	-	231	336	-	200	1.79	-	1.07
8	0.65	$ 1,060	$ 13.00	$ 2.03	2.97	33,674	8,472	25,201	2,322	6,150	-	460	-	274	398	-	236	1.69	-	1.00
9	0.70	$ 1,141	$ 14.00	$ 2.19	3.12	38,166	9,270	28,896	2,397	6,873	-	491	-	291	425	-	251	1.65	-	0.98
10	0.75	$ 1,223	$ 15.00	$ 2.35	3.23	40,813	9,639	31,174	2,466	7,173	-	506	-	300	437	-	259	1.63	-	0.97
11	0.80	$ 1,304	$ 16.00	$ 2.50	3.30	42,859	9,968	32,891	2,492	7,476	-	518	-	306	448	-	264	1.62	-	0.96
12	0.85	$ 1,386	$ 17.00	$ 2.66	3.75	50,280	10,582	39,698	2,545	8,037	-	547	-	316	472	-	273	1.61	-	0.93
13	0.90	$ 1,467	$ 18.00	$ 2.82	4.03	55,924	11,115	44,810	2,588	8,526	-	569	-	326	490	-	281	1.59	-	0.91
14	0.95	$ 1,549	$ 19.00	$ 2.97	4.72	71,794	12,546	59,249	2,610	9,936	-	627	-	341	538	-	294	1.55	-	0.85
15	**1.00**	**$ 1,630**	**$ 20.00**	**$ 3.13**	**4.96**	78,170	13,117	65,053	2,612	10,505	-	649	-	346	556	-	298	**1.54**	**-**	**0.82**
16	1.05	$ 1,712	$ 21.00	$ 3.29	5.13	81,884	13,349	68,535	2,623	10,726	-	659	-	349	565	-	301	1.54	-	0.81
17	1.10	$ 1,793	$ 22.00	$ 3.44	5.34	86,289	13,614	72,675	2,630	10,984	-	670	-	351	574	-	303	1.53	-	0.80
18	1.15	$ 1,875	$ 23.00	$ 3.60	5.52	90,669	13,905	76,763	2,631	11,274	-	682	-	355	584	-	306	1.52	-	0.79
19	1.20	$ 1,956	$ 24.00	$ 3.76	5.65	93,454	14,043	79,411	2,641	11,402	-	688	-	356	589	-	307	1.52	-	0.79

16.3.4 Design Criteria

Table 16-10 shows the final pit ramp design parameters.

Table 16-10: Final Pit Ramp Design Parameters

Parameter	Units	Main/NW	Main/NW	SE	SE	Sona Hill	Sona Hill
Material Type		Saprolite	Fresh	Saprolite	Fresh	Saprolite	Fresh
Maximum Ramp Width	m	27	27	27	27	27	27
Number of ramps per wall		Variable	Variable	Variable	Variable	Variable	Variable

Source: SRK, 2021

16.3.4.1 Geotechnical Parameters

Table 16-11 shows the geotechnical parameters used for the PEA pit design. Due to no changes in the geotechnical parameters, SRK applied the same inter-ramp angles to the new pit design. Table 16-12 shows the pit geotechnical parameters for the saprolite and fresh rock. SRK used the PEA geotechnical parameters and applied smoothing between the proposed inter-ramp angles to ensure that smooth sector transitions.

Table 16-11: Toroparu Pit Final Geotech Pit Design Parameters Used in PEA

Rock Type	Pit Design Sector	Pit Wall Orientation (°)	Kinematic Failure Mode	Bench Face Angle (°)	Bench Height (m)	Bench Width (m)	Inter-ramp Angle (°)
Saprolite		-	-	65	10	8	38
Fresh Bedrock	Northeast	220	Planar	65	20	9.5	45
	East	250	Planar	70	20	9.5	50
	South	355	Toppling	75	20	10	53
	Southwest	40	Toppling	75	20	10	53
	Northwest	120		75	20	10	53

Source: Knight Piésold. *The geotech parameters above do not include the SE and Sona Hill Pit areas. SRK used 45° inter-ramp angles for the SE and Sona Hill Pit designs.

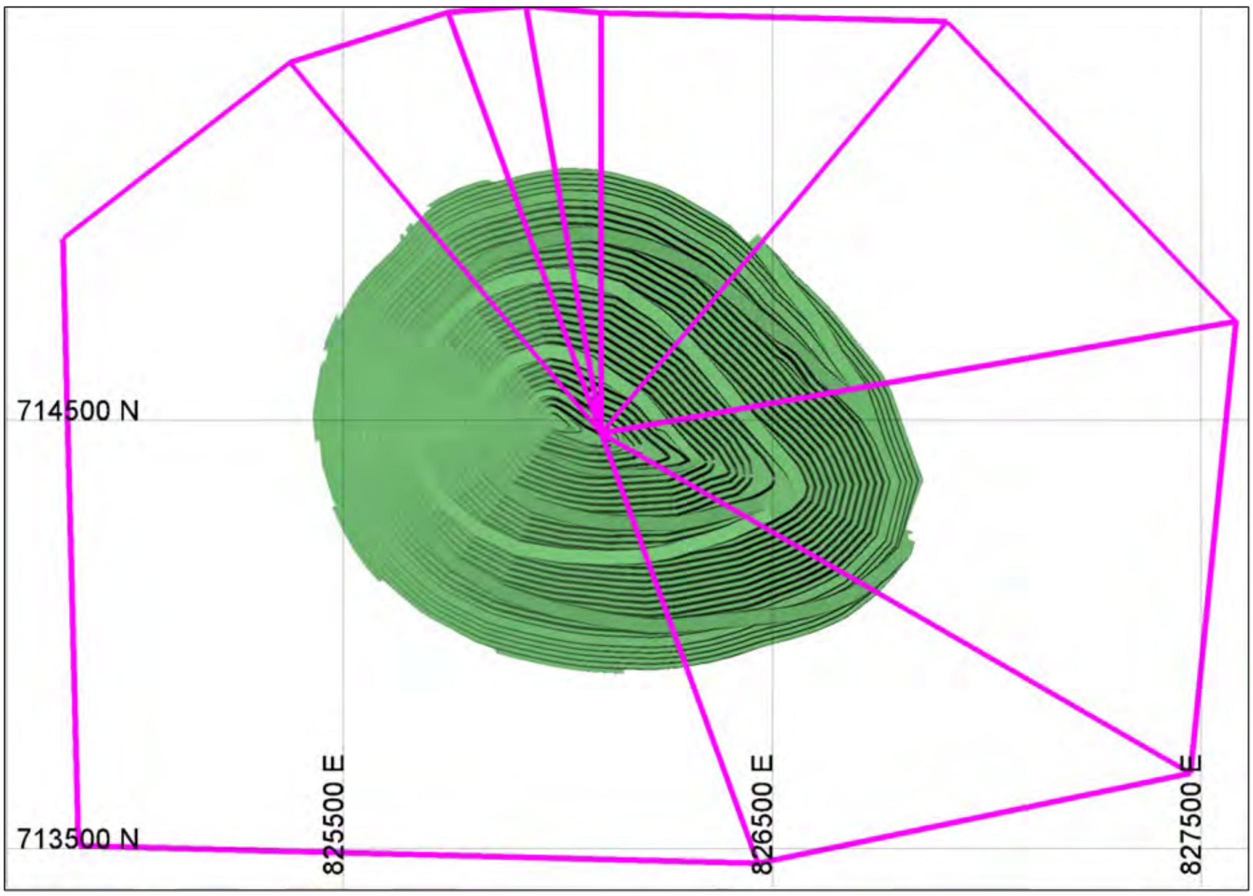
Table 16-12: Toroparu and SE Pits Geotech Pit Design Parameters

Pit	Rock Type	Start Azimuth	End Azimuth	Berm Width (m)	Batter Angle (º)	Bench Height	Double Bench	Inter-ramp Angle (º)
Main/NW	Fresh	0	5	10.00	70	20	YES	49
		5	10	11.25	70	20	YES	47
		10	15	11.75	70	20	YES	47
		15	20	12.25	70	20	YES	46
		10	45	12.75	70	20	YES	45
		45	48	12.25	70	20	YES	46
		45	50	11.75	70	20	YES	47
		50	55	10.75	70	20	YES	48
		55	60	10.00	70	20	YES	49
		60	115	9.50	70	20	YES	50
		115	120	9.00	70	20	YES	51
		120	125	8.50	70	20	YES	52
		125	130	8.20	70	20	YES	52
		130	345	7.80	70	20	YES	53

Pit	Rock Type	Start Azimuth	End Azimuth	Berm Width (m)	Batter Angle (º)	Bench Height	Double Bench	Inter-ramp Angle (º)
		345	350	9.00	70	20	YES	51
		350	355	9.25	70	20	YES	50
		355	0	9.75	70	20	YES	50
Main/NW	Saprolite	0	0	9.25	70	10	NO	37
SE	Fresh	0	0	12.75	70	20	YES	45
SE	Saprolite	0	0	9.25	70	10	NO	37
Sona	Fresh	0	0	4.9	70	10	NO	50
Sona	Saprolite	0	0	9.25	50	10	NO	37

Source: SRK, 2021

Figure 16-3 shows the Main/NW Pits geotechnical sectors used to design the phase designs and final pit design. Blending sectors were added to ensure smooth transition between 45° to 53° sectors. Saprolite material was excluded from this methodology.



Source: SRK, 2021

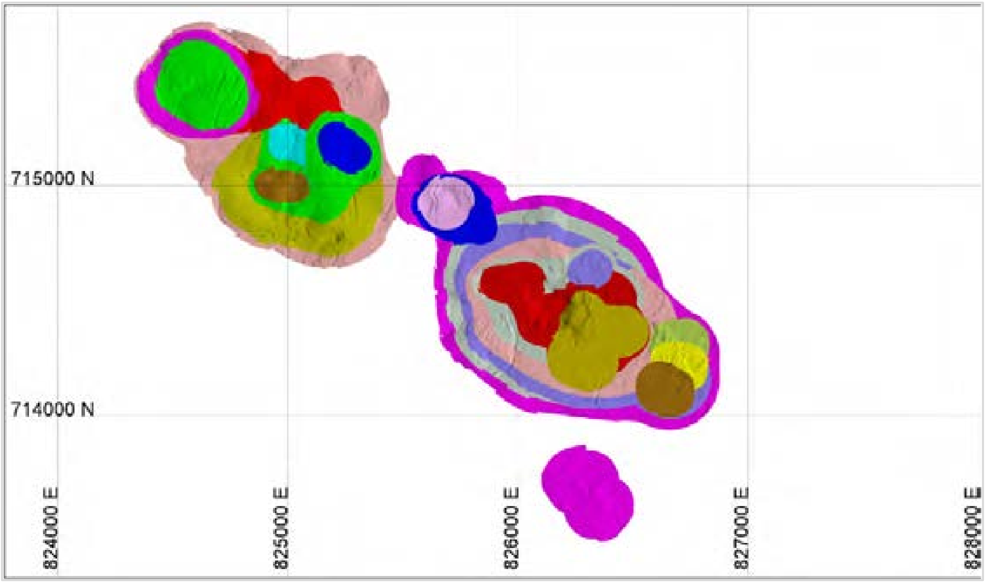
Figure 16-3: Main/NW Pit geotech sectors used for pit design

16.3.5 Pit Phase and Ultimate Pit Designs

To ensure proper economic material exposure and access to different Au grades, SRK created multiple mining phases. The Main and NW Pits, which are located towards the north of the deposit, contains 13 mining phases. The SE Pit, which is located in the south part of the property contains four mining phases.

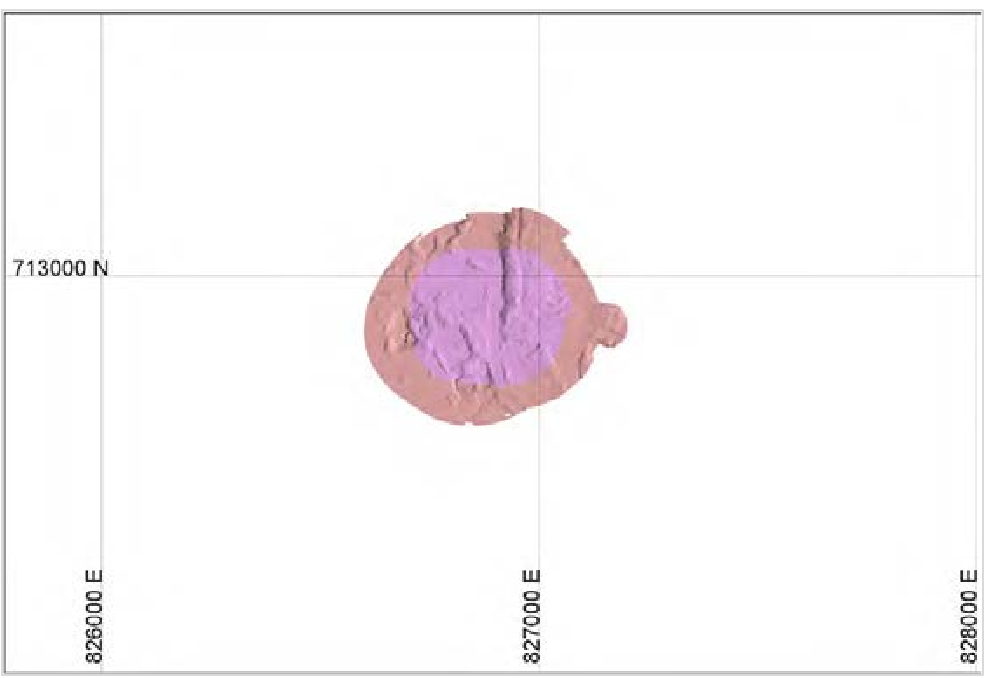
To improve the economics of the Project, phases were divided following pit optimization shells to ensure that the higher profit pit shells were being mined first. Pit optimization shells were also selected based on the equipment size to minimize the need for wider access roads which would increase the initial capital costs. Based on this information, a US$900/oz Au value pit shell was chosen for detailed phase design.Source: SRK, 2021

Figure 16-4 shows the Main and NW Pit phase designs. Figure 16-5 shows the SE Pit phases. Figure 16-6 shows the Sona Hill Pit phases.



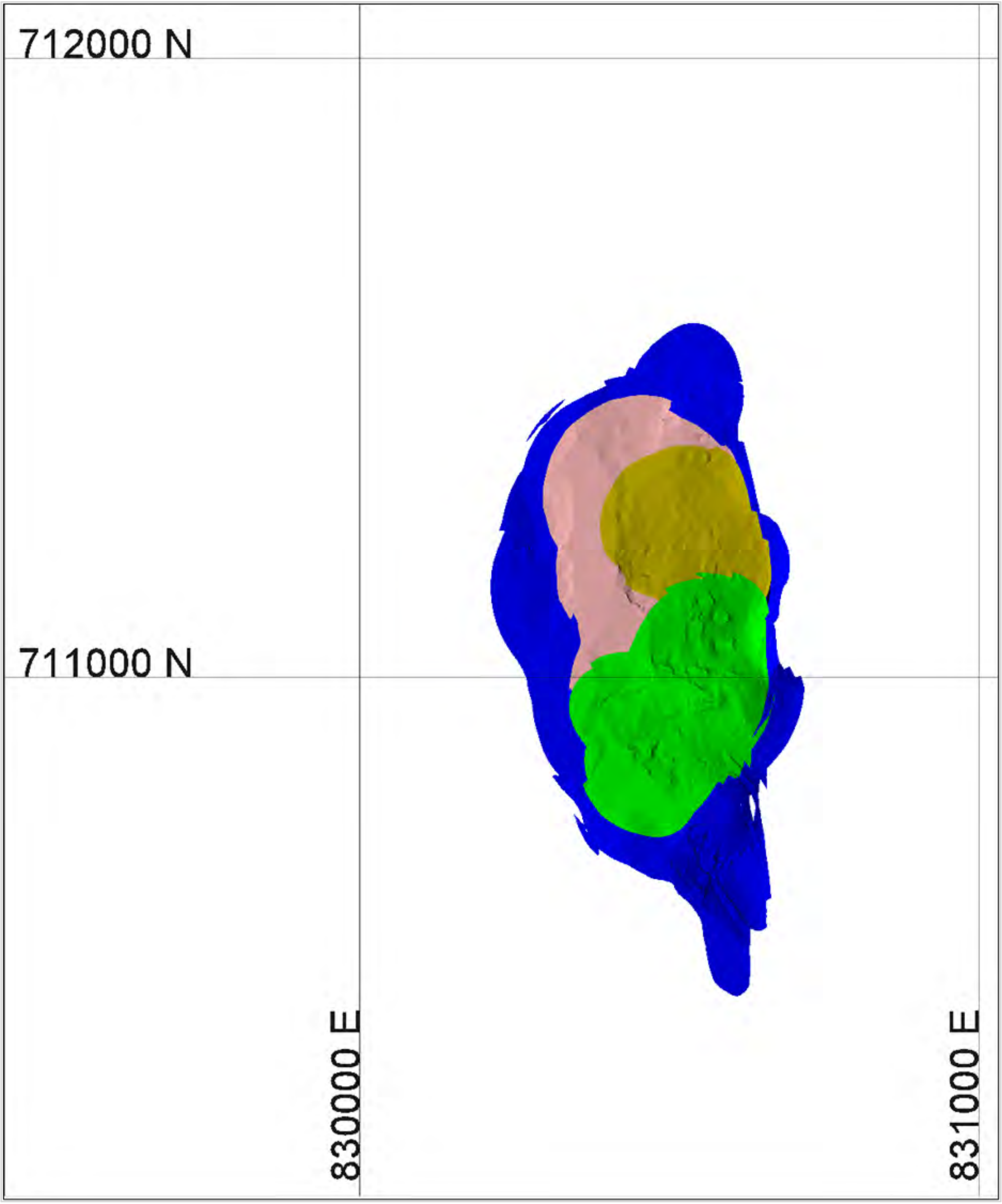
Source: SRK, 2021

Figure 16-4: Toroparu Main, NW Pit phase designs



Source: SRK, 2021

Figure 16-5: SE Pit phase designs



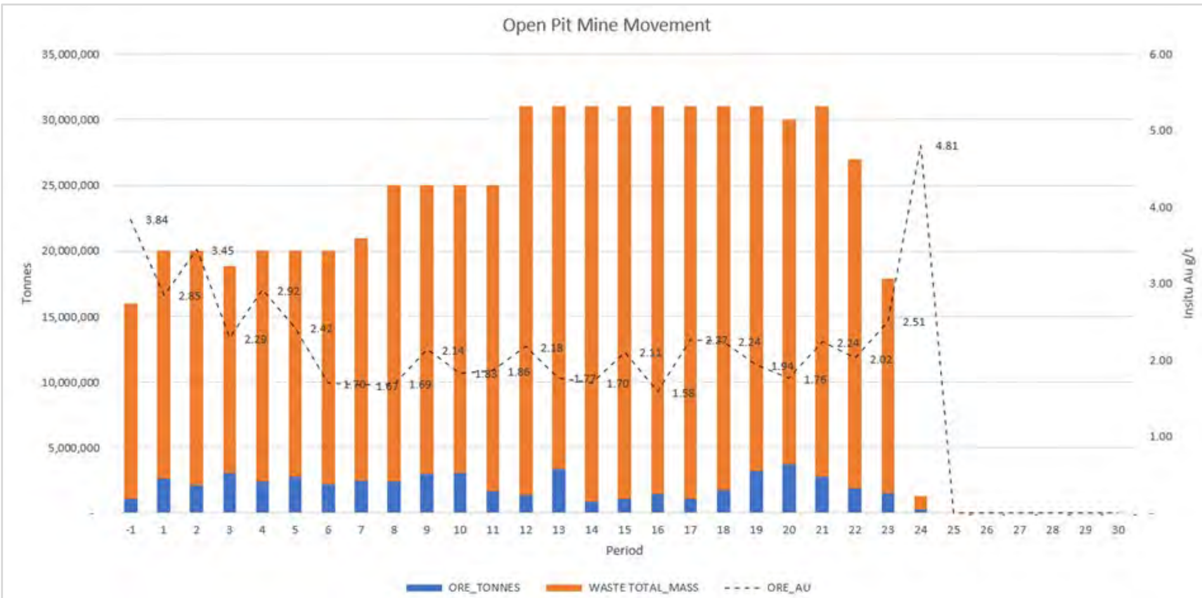
Source: SRK, 2021

Figure 16-6: Sona Hill Pit phase designs

16.3.6 Mine Production Schedule

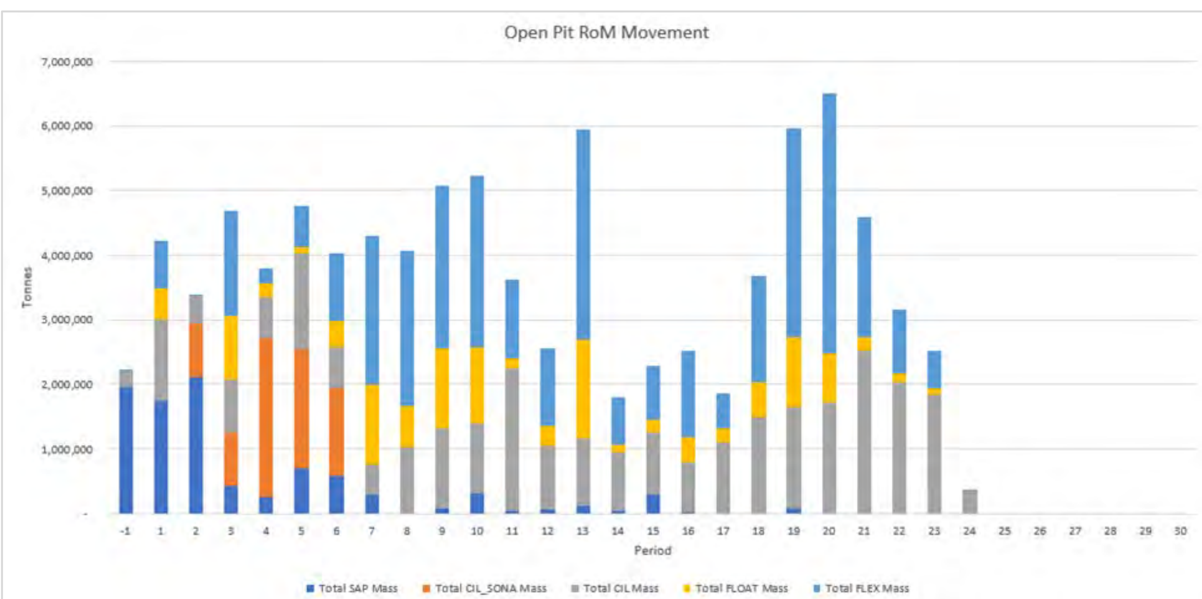
Au production of at least 200,000 oz per year was targeted at the request of the Company, depending on the material feed. No hard-maximum material movement limit was placed on the production schedule.

The operation is planned to be mined using an owner operated mining fleet. Therefore, a key feature of the mining schedule is to ensure a practical mining fleet configuration could be maintained throughout the life of the Project. Figure 16-7 through Figure 16-11 show the combined open pit and underground mine production schedule for material moved, stockpiles, mine production and head grade.



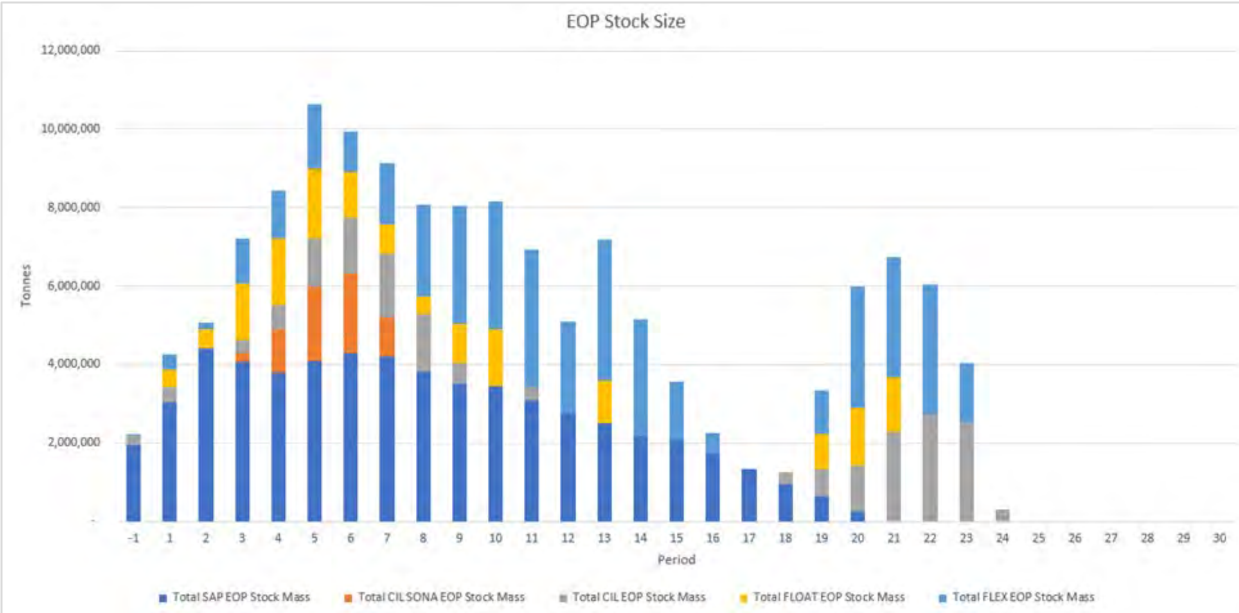
Source: SRK, 2021

Figure 16-7: Combined open pit and underground direct mine movement



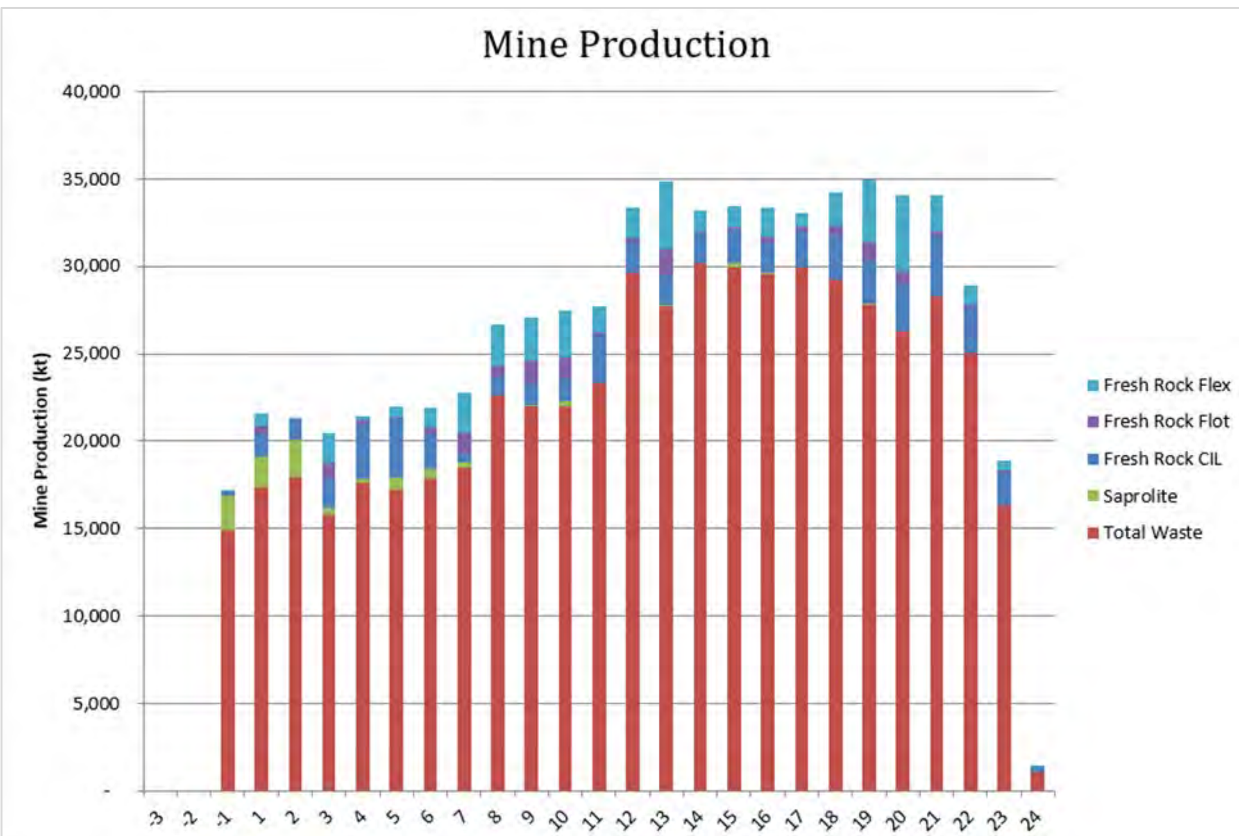
Source: SRK, 2021

Figure 16-8: Combined open pit and underground direct movement by material type



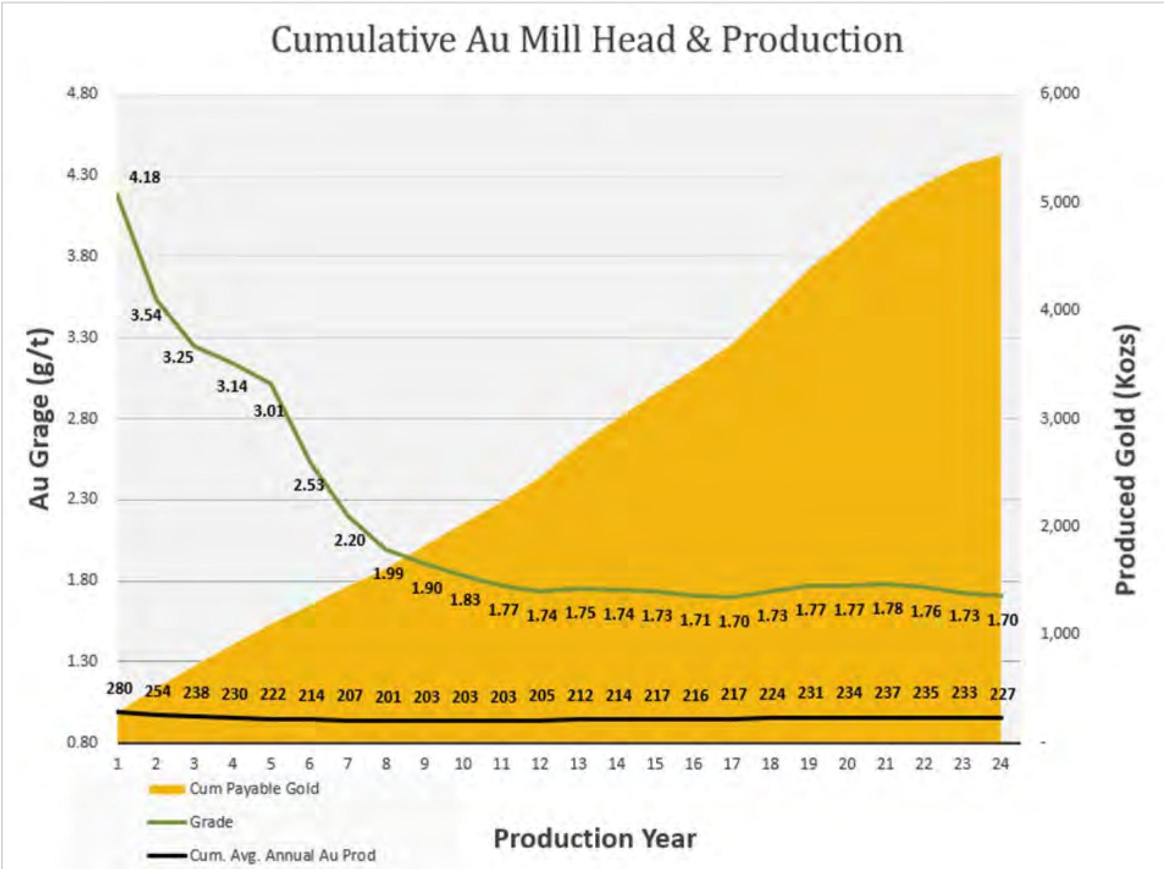
Source: SRK, 2021

Figure 16-9: End of year stockpile tonnes for open pit and underground combined



Source: SRK, 2021

Figure 16-10: Open pit and underground mine production combined



Source: SRK, 2021

Figure 16-11: Open pit and underground cumulative mill Au grade and production

16.3.6.1 Grade Control

Grade control will be very important to ensure that only the higher-grade materials are being sent directly to the crusher. Grade control process for the 5 m benches (mined by the large equipment fleet from Year 7 on) will be as follows:

- All blastholes will be sampled near the mineralized zones;
- Since the mining bench height is set to 5 m for mining with the large equipment, an “A and B” sampling technique will be developed for areas where high and low-grade material is to be expected. The A and B sample works as follows:
  - Drillers/Samplers will gather the top 2.5 m of drill cuttings and will define as the A sample. The B sample will be the lower 2.5 m of drill cuttings plus sub-drill; and
  - A and B samples will be analyzed in a laboratory setup on site.
- Areas where the grades are known to be constant will not need the A and B sample technique;
- The geologist will estimate separately the A and B samples for each pattern to determine if a split bench mining using 2.5 m flitches will result in better economics. If not, the A and B samples will be mined together at the full 10 m bench height. Wherever possible, the split bench option will be evaluated; and
- The geologist and surveyors will place flags in the pattern based on the grade control outlines.

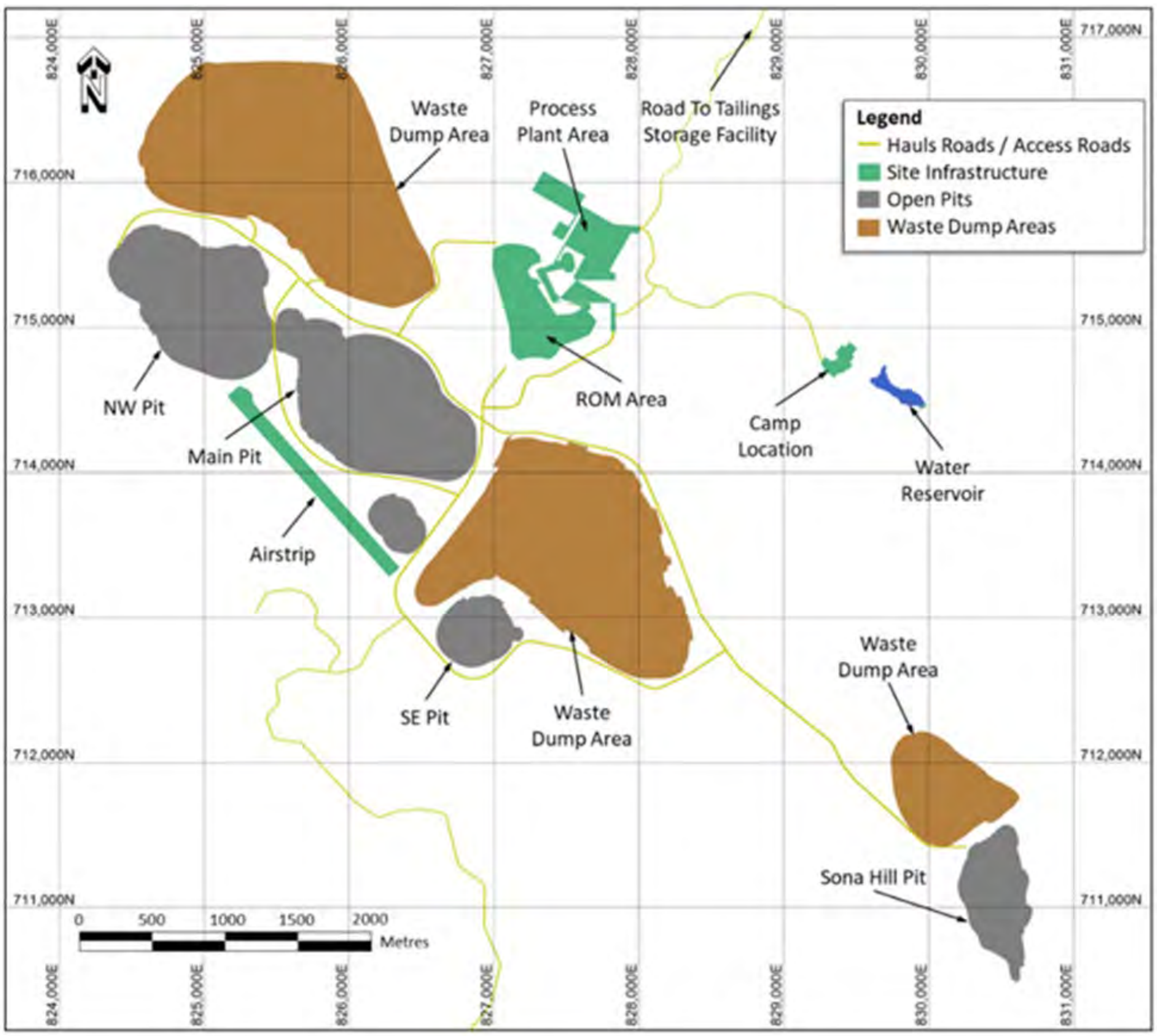
16.3.7 Waste and Stockpile Design

16.3.7.1 Waste Rock Storage Facility

The waste rock storage for the Toroparu Deposit operation has been designed to limit the vertical expansion of the waste dump and have dump toes located for control of surface run-off. The dumps have also been located in areas that that will not be impacted by potential future mining operations. The dumps are not placed on the west side of the pit, which is mineralized and where future mining operations may occur.

Waste rock produced from the Toroparu Deposits mining operations (Main, NW and SE Pits) will be placed on existing terrain in two designated areas. The East Dump is located between the east of the final Main and NW Pits and north of the SE Pit. The North Dump is to the north of the Main and NW Pits creating a natural shield (levee) from extreme rain events. The South East Backfill Dump is within the mined out SE Pit area. This dump will be used in the last two years of mining to help shorten the haul truck cycles.

Figure 16-12 shows the pits and waste dumps for the full site area.



Source: SRK, 2021

Figure 16-12: Site map

Scheduling of waste rock removal from the open pit will facilitate material management and will allow for the waste pile to be segmented as required.

The closure activities will include construction of a series of catchment and diversion ditches to collect surface run-off from the waste rock storage areas. The ditches will be designed to control sediment loading into the natural water drainage, to minimize erosion of surface materials, and to divert any metal-rich waters as required.

The dump designs follow industry standard design with an overall angle of 21° to 23°. Due to high quantities of rain, SRK decided to make the waste dump angles shallower. Table 16-13 shows the waste dump parameters used for the design.

Table 16-13: Waste Dump Parameters

Parameter	Unit	Value
Overall Slope Angle	°	21
Batter Angle	°	36
Bench Height	m	10
Berm Width	m	13.5
Ramp Width 2 way	M	30
Ramp Width 1 way	M	18
Ramp Gradient	%	9

Source: SRK, 2021

16.3.7.2 Ore Stockpiles

The mine plan relies on the creation of low-grade stockpiles near the crusher location which will be used when insitu economic material is not readily available, and to continue feeding the plant after the end of the pit mining. The economic material stockpile will be located north of the East Dump and south of the processing plant site. Surface area for the economic material stockpile is estimated to be 411,848 m^2 with a volume capacity of 10.3 Mm^3 or close to 20 Mt of stockpile economic material grade material. The stockpile area can be expanded to hold double the current capacity with minimal costs.

The economic material stockpile creation and planning will be based on the following procedures:

- Waste fresh rock material will create the base of the economic material stockpile. The surface area is flat so minimal waste material will be required;
- In the early years of the mine schedule, material with Au grades above 1.0 g/t will be shipped directly from the pit to the crusher. Material between 0.35 g/t and 1.0 g/t Au will be stockpiled temporarily. It is estimated that five to six different grade bins (stockpiles) will be needed to ensure that the highest Au grade material is sent directly to the crusher, while the lower grade material is stockpiled;
- All stockpiled material above 1.0 g/t Au will only remain in the stockpile for less than one year, while material between 0.5 g/t and 1.0 g/t Au will remain longer, with some being processed in the last two years of the mill operations;
- The stockpile will be mined by small wheel loaders in combination with 41 t haul trucks (articulated dump trucks [ADT]). A total of 21 Mt of low-grade will need to be rehandled through the LoM; and
- The stockpile pad will be built such that all water drainage will be accumulated in the north-west area, and water pumps will be placed to discharge water.

16.3.8 Mining Fleet and Requirements

Mining methods will be open pit mining using hydraulic excavators and wheel loaders loading articulated dump trucks for waste and economic material haulage. The operations are described further in the following sections.

Mining activities at the Toroparu and Sona Hill mining operations will include removal of growth medium (topsoil), free-digging, drilling, blasting, loading, hauling and mining support activities. Material within the pits will be generally blasted on a 5 m high bench, at least until the mining operations expand at the Main and NW Pits in Year 10. Saprolite material can be loaded directly with hydraulic excavators without the need for blasting. Waste dumps will be used for material below the cut-off grade, and stockpiles for lower grade economic material above the cut-off grade. Lower grade economic material from all pits will be placed in stockpiles, near to the primary crusher location. Higher-grade economic material will be sent directly to the primary crusher.

16.3.8.1 Mining Requirements and Fleet Selection

Specific requirements dictated the selection of mining equipment types and sizes. Loading equipment selection focused on generally having diesel-powered (track-driven) hydraulic excavators, together with front-end wheel loaders available for added operational flexibility.

Hydraulic excavators will be primarily used for loading in the open pits, with the front-end loaders primarily used for loading in the low-grade stockpile and some loading in the pits. Trucks will be matched to the loading equipment units. Additional equipment units were provisioned when required, in keeping with the planned mine production schedule requirements. Allowances were made for swapping of equipment between the Toroparu and Sona Hill mining operations and use of certain support equipment at both locations.

The major mine equipment fleet requirements were based on the annual mine production schedule, the mine work schedule, and shift production estimates. The mine equipment requirements and costing were based on the purchase of new equipment. The equipment fleet selection and requirements are further discussed in the individual sections that follow in this report.

It was planned that all mine mobile equipment would be diesel-powered to avoid the requirement to provide electrical power into the pit working areas.

The mine operations schedule is proposed to include two twelve-hour shifts per day, seven days per week for 355 days per year, which includes an annual allowance of 10 days downtime for weather delays for most of the mine operations, and 15 days downtime for weather delays for the drilling operations. Mine productivity and costing included estimating the productive operating time per twelve-hour shift. Non-productive time per shift includes shift change (travel time), equipment inspections, fuelling, and operator breaks. It was estimated that the total time per shift for these items will be 2.0 hours. The scheduled production time (scheduled operating hours) was therefore estimated at 10.0 hours per shift, representing a (shift) utilization of 83% of the twelve-hour shift period (and excludes mechanical availability and work efficiency factors).

In addition, allowances were made for work efficiencies including equipment moves (production delays while moving to other mining areas within the pit), and certain dynamic operational inefficiencies. These work efficiencies are further detailed in the respective sections for drilling, loading and hauling.

Equipment fleet mechanical availability was estimated for the various major mine equipment fleets, including drills (75%), hydraulic excavators (85%), front-end loaders (80%), trucks (85%), etc. (with replacement equipment units assumed to be new).

Table 16-14 shows the mining equipment requirements for selected years of the mine plan and includes the mining operations at Toroparu and Sona Hill. Years 23 and 24 involve only stockpile re-handling operations (with no pit mining).

Table 16-14: Planned Mining Equipment Requirements for Selected Years

Equipment Units Used	Make	Model	Size	-1	1	2	3	4	6	8	9	10	12	14	16	18	20	22	23	24
Drilling																				
Blasthole drill-new	AtlasCopco	D65LF	152 mm	2	3	3	3	3	3	4	4	4	4	5	5	5	6	2	-	-
Loading																				
Front-end loader-new	Caterpillar	988K	6.4 m^3	2	3	3	3	3	4	4	4	5	5	4	4	4	4	4	-	-
Front-end loader-new	Caterpillar	993K	12.2 m^3	-	-	-	-	-	-	-	-	1	1	1	1	1	1	1	1	-
Hydraulic excav-new	Caterpillar	390FL	5.7 m^3	4	5	6	6	6	6	6	6	4	4	4	3	3	3	2	-	-
Hydraulic excav-new	Caterpillar	6040EX	22.0 m^3	-	-	-	-	-	-	-	-	1	2	3	3	3	3	2	2	2
Hauling																				
Haul truck-new	Caterpillar/Volvo ADT	N/A	58t	18	26	24	26	28	27	4	4	4	4	4	4	4	4	4	4	4
Haul truck-new	Scania	G460CB10X4	50t	2	2	2	2	2	2	2	2	2-	2	2	2	2	2	2	2	2
Haul truck-new	Caterpillar	777D	91t	0-	0	0	0	0	0	14	16	20	23	29	37	37	37	32	30	25
Other Mine Equipment																				
Crush/Screen Plant	Manufacturer	Jaw/Cone	335 kW	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Track dozer-new	Caterpillar	D9T	306 kW	4	4	5	5	5	5	5	5	4	4	4	4	4	4	4	2	2
Wheel dozer-new	Caterpillar	834K	419 kW	2	2	2	2	2	2	2	2	1	1	1	1	1	1	1	-	-
Wheel dozer-new	Caterpillar	844H	468 kW	-	-	-	-	-	-		-	1	1	2	2	2	2	1	1	1
Motor grader-new	Caterpillar	16 M3	216 kW	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	1	1
Backhoe loader-new	Caterpillar	450F	102 kW	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Water truck-new	Scania	P410CB8X4	30,000 L	2	2	2	2	2	2	2	3	2	2	2	3	3	2	2	1	1
Excavator-new	Caterpillar	374FL	352 kW	2	2	2	2	2	2	2	2	1	1	1	1	1	1	1	1	1
Compactor-new	Caterpillar	CS/CP74	116 kW	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1

Equipment Units Used	Make	Model	Size	-1	1	2	3	4	6	8	9	10	12	14	16	18	20	22	23	24
Support Equipment																				
Transport/mover	Manufacturer	Model	136t	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Truck crane	Manufacturer	Model	40t crane	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Recovery truck	Scania	G460CB8X8	360 kW	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Secondary blast drill	Manufacturer	75 kW	64 mm	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Fuel truck	Scania	P410CB8X4	30,000 L	2	2	2	2	2	2	2	3	3	2	2	3	3	3	2	1	1
Lube truck	Scania	P410CB8X4	30,000 L	2	2	2	2	2	2	2	3	3	2	2	3	3	3	2	1	1
HD mechanic's truck	Scania	P360CB6X4		2	2	2	2	2	2	2	3	3	2	2	3	3	3	2	1	1
Welding truck	Scania	P360CB6X4		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Tire service truck	Scania	P360CB6X4		1	1	1	1	1	1	1	1	2	2	2	2	2	2	1	1	1
Forklift	Manufacturer	Model		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Flatbed truck	Scania	P360CB6X4	19t crane	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Personnel van/bus	Manufacturer	Model		3	3	3	3	3	3	3	3	4	4	4	4	4	4	2	2	2
Service pick-up	Manufacturer	4x4		18	18	18	18	18	18	18	18	15	15	15	18	15	15	15	10	10
Light plant	Manufacturer	Portable	8 kW	21	21	21	21	21	21	21	21	20	20	20	20	20	20	20	15	15
Blasting																				
Blasting flatbed truck	Scania	G360CB4X4		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-
ANFO/Emulsion truck	Scania	P360CB6X4	13t	1	1	1	2	2	2	2	2	2	2	2	2	2	2	1	-	-
Blaster screw truck	Manufacturer	4x4		1	1	1	2	2	2	2	2	1	1	1	1	1	1	1	-	-
Blasthole stem truck	Scania	P360CB6X4		1	1	1	2	2	2	2	2	2	2	2	2	2	2	1	-	-

Source: SRK, 2021

16.3.9 Drilling

The planned drilling equipment fleet will consist of Atlas Copco D65LF units. This fleet was based on drilling 152 mm blastholes to an average depth of 5.75 m (including a 0.75 m sub-drill) for development of 5 m high benches. The drills can single-pass drill (no rod changes) such holes. From Year 10 on, 10 m high benches will be mined for the large equipment fleet, so the drilling method will be changed at that time.

The planned nominal production blasthole pattern is equivalent to a 5.75 m x 5.75 m pattern (spacing and burden) in waste and economic material, however, in practice the burden and spacing will vary. (The planned nominal 5.75 m square pattern would be approximately equivalent to a 5 m x 7 m pattern.) For the main production drilling an instantaneous drilling rate of 0.60 m/minute was estimated for waste and economic material. Allowances were made in the drilling productivity estimates for re-drills and additional control blasting requirements (15%) and moving to new working areas. Fleet requirements were based on drilling all of the fresh rock within the planned open pits, and for grade control purposes all saprolite material and 25% of saprolite waste (to determine the economic material limits).

Table 16-15 shows selected drilling statistics based on the planned drilling equipment and drilling patterns for waste and economic material at Toroparu.

Table 16-15: Drilling Statistics Per Unit

Item	Unit	Value
Rock Type		Waste and Ore
Waste/Ore Pattern Size	m^2 x m^2	5.75 x 5.75
Drilling Tram and Set Up Time	min/op hr	13.4
Drilling Penetration Rate	m/min	0.60
Drilling Time per Blasthole	min	9.6
Moving and Delay Time	min/op hr	10
Production per Unit (100% Available) *	t/op hr	1,480

Source: SRK, 2019. * Includes allowance of 15% for re-drills and control blasting patterns.

Table 16-16 shows selected drilling productivity information based on the planned drilling equipment at Toroparu. Annual production capacity for per drill is 7.8 Mtpa.

Table 16-16: Drilling Productivity Per Unit

Item	Unit	Value
Rock Type		Waste and Ore
Production per Unit (100% Available)	t/op hr	1,480
Planned Operating Hours per Shift	scheduled op hrs	10.00
Planned Operating Hrs per Year *	scheduled op hrs	7,000
Estimated Mechanical Availability **	%	75%
Actual Operating Hours per Year	op hrs	5,250
Annual Production Capacity per Unit	Mtpa	7.8

Source: SRK, 2019. * Includes allowance of 15 days downtime for weather delays. ** Typical mechanical availabilities for drills used.

16.3.10 Blasting

Bulk emulsion explosives will be used for blastholes. Blasting requirements were based on blasting all fresh rock within the planned open pits.

The powder factor for production blasting was estimated to be 0.205 kg/t (kg explosives per tonne of rock), based on an estimate by Orica Mining Services. As previously mentioned, a 15% contingency allowance was made for additional blasthole drilling (closer drilling to achieve control blasting and for proper fragmentation), and this contingency also includes the necessary explosives.

At some stage in the Project, the explosives provider for the mine will have a dedicated bulk emulsion plant, which will be capable of sufficient production for the planned mining operations. Prior to that bulk emulsion can be shipped to site in 25-ton isotanks. Blasting accessories will be transported to site and stored in suitable explosives magazines.

The mine will initially have a 13-ton emulsion truck, which will deliver bulk explosives to the blast sites during daylight hours. The blasting equipment fleet will initially include a dedicated stemming truck, a flatbed truck and blasting crew truck. Stemming material will be mainly drill cuttings. The mine blasting crew will manage and conduct the blasting operations.

16.3.11 Loading

Loading equipment selection included having a combination of diesel-powered hydraulic excavators and front-end loaders for operational flexibility. The hydraulic excavators are capable of mining more selectively and will be used for most of the pit mining. The front-end loaders will be used primarily for loading in the low-grade stockpile and some loading in the pits

The loading equipment fleet for the earlier years of the mining operations was planned to be a combination of equipment consisting of up to seven smaller hydraulic excavators (5.7 m^3 Caterpillar 390 FL class, Excav1), and up to four front-end loaders (6.4 m^3 Caterpillar 988K class loaders, FEL1). This equipment will load a fleet of 41-tont capacity articulated dump trucks (Volvo A45G class ADTs). A fleet of up to 36 ADTs was planned for earlier years (up to Year 13) with a fleet of 16 thereafter. In addition to pit mining, these units will perform re-handling of economic material from the low-grade stockpiles to the primary crusher (except for Years 23 and 24 when the pit mining has finished and only stockpile re-handling is taking place with larger trucks).

Starting in Year 10 a larger loading equipment unit will be brought into operation. By Year 11 these larger units will consist of two hydraulic excavators (22.0 m^3 Caterpillar 6040 class, Excav2), and one large front-end loader (12.2 m^3 Caterpillar 993K class, FEL2). These units, reaching a maximum fleet of three in Year 14, will load a fleet of 132-ton capacity haul trucks (Caterpillar 785G class units).

At Sona Hill economic material hauled by ADTs will be placed in a re-handling area near the pit. The economic material will be re-loaded by a 6.4 m^3 Caterpillar 988K class loader into two 50-ton capacity trucks (Scania G460CB 10 x 4 class units) for transport to the Toroparu main complex for delivery to either the primary crusher, or to one of the low-grade stockpiles. Presently, it is planned that economic material will be rehandled and hauled to the main Toroparu complex at the time it is mined from Sona Hill.

The hydraulic excavators will be able to free-dig approximately the saprolite material within the planned open pit. Within the Toroparu Deposit the dry density for saprolite was estimated to be 1.84 t/m^3 and for fresh rock 2.76 t/m^3. Within the Sona Hill deposit the dry density for saprolite was estimated to be 1.66 t/m3 and for fresh rock 2.84 t/m^3. Saprolite moisture content was estimated to be 20% on average (varying with season and depth) and swell in loading to be 20%. Fresh rock moisture content was estimated to be 6% on average and swell in loading to be 40%.

Table 16-17 shows selected loading statistics for the planned loading units in saprolite waste at Toroparu.

Table 16-17: Loading Statistics by Unit Type in Saprolite Waste at Toroparu

Equipment Type	Unit	Hyd Excav1	Hyd Excav2
Av. Bucket Size and Fill Factor	m^3, %	5.7, 72%	22.0, 71%
Matched Truck Rated Size	t	41	132
Number of Passes		5	5
Total Truck Loading Time *	**min**	**3.13**	**3.34**
Moving and Delay Time	min/op hr	13	14
Ore Prod. per Unit (100% Available)	dry t/op hr	511	1,647

Source: SRK, 2021. * Includes truck spotting time and 90% operator efficiency. Average 20% moisture assumed.

Table 16-18 shows selected loading statistics for the planned loading units in fresh rock waste at Toroparu.

Table 16-18: Loading Statistics by Unit Type in Fresh Rock Waste at Toroparu

Equipment Type	Unit	FE Loader1	Hyd Excav1	FE Loader2	Hyd Excav2
Av. Bucket Size and Fill Factor	m^3, %	6.4, 77%	5.7, 69%	12.2, 86%	22.0, 72%
Matched Truck Rated Size	t	41	41	132	132
Number of Passes		4	7	6	4
Total Truck Loading Time *	min	2.69	3.13	4.29	2.73
Moving and Delay Time	min/op hr	12	14	12	14
Ore Prod. per Unit (100% Available)	dry t/op hr	692	569	1,388	2,105

Source: SRK, 2021. * Includes truck spotting time and 90% operator efficiency. Average 6% moisture assumed.

The total truck loading times included a truck spotting (initial positioning of the trucks for loading) time of 42 seconds.

Table 16-19 shows selected loading productivity information based on the planned loading equipment in saprolite waste at Toroparu.

Table 16-19: Loading Productivities by Unit Type in Saprolite Waste at Toroparu

Equipment Type	Unit	Hyd Excav1	Hyd Excav2
Ore Prod. per Unit (100% Available)	dry t/op hr	511	1,647
Planned Operating Hours per Shift	scheduled op hrs	10.0	10.0
Planned Operating Hours per Year *	scheduled op hrs	7,100	7,100
Estimated Mechanical Availability **	op hrs %	85%	85%
Actual Operating Hours per Year	op hrs	6,035	6,035
Annual Economic Material Production Capacity per Unit	dry Mtpa	3.1	9.9

Source: SRK, 2021. * Includes allowance of 10 days downtime for weather delays. ** Typical mechanical availabilities for excavators used.

Table 16-20 shows selected loading productivity information based on the planned loading equipment in fresh rock waste at Toroparu.

Table 16-20: Loading Productivities by Unit Type in Fresh Rock Waste at Toroparu

Equipment Type	Unit	FE Loader1	Hyd Excav1	FE Loader2	Hyd Excav2
Ore Prod. per Unit (100% Available)	dry t/op hr	692	569	1,388	2,105
Planned Operating Hours per Shift	scheduled op hrs	10.0	10.0	10.0	10.0
Planned Operating Hours per Year *	scheduled op hrs	7,100	7,100	7,100	7,100
Estimated Mechanical Availability **	op hrs %	80%	85%	80%	85%
Actual Operating Hours per Year	op hrs	5,680	6,035	5,680	6,035
Annual Economic Material Production Capacity per Unit	dry Mtpa	3.9	3.4	7.9	12.7

Source: SRK, 2021. * Includes allowance of 10 days downtime for weather delays. ** Typical mechanical availabilities for excavators, shovels and loaders used.

As part of the mining operations, an allowance was made for re-handling 2% of the plant economic material feed in a stockpile adjacent to the primary crusher with a loader only. Re-handling of the main stockpile economic material was included in the loading/hauling operations. Additional loading operations by the smaller front-end loaders included crushed waste backfill to be hauled to the pits for roadway surfacing.

16.3.12 Hauling

The truck sizes selected were determined by loading unit/truck matching, maintaining the necessary degree of operational flexibility, and meeting production requirements.

Waste will be hauled to the waste dumps. Economic material will be hauled either to the primary crusher or main economic material stockpile area. Economic material at Sona Hill will be rehandled near the pit by a loader into two 50-ton capacity trucks (Scania G460CB 10 x 4 class units) and then hauled to the main Toroparu complex at the time it is mined from Sona Hill.

The main hauling equipment fleet for the earlier years of the mining operations was planned to be comprised of 41-ton capacity ADTs (Volvo A45G class units). This type of unit is commonly used in saprolite mining. A fleet of up to 36 ADTs was planned for mining haulage up to Year 13 of mining production with a fleet of 16 thereafter.

A fleet of up to two 50-ton capacity trucks (Scania G460CB 10x4 class units) was planned for hauling economic material from Sona Hill to the main Toroparu complex (with three units in Year 9).

Starting in Year 10 when the mining operations are expanded, larger mining equipment will be introduced into the operations including 132-ton capacity rear dump trucks (Caterpillar 785D/G class units). A fleet of up to 20 units was planned for operations up to Year 20, and the maximum 785D/G fleet reaches 22 units in Year 21.

Stockpile re-handling haulage was planned to be performed by ADTs up to and including part of Year 23. For Years 23 and 24 stockpile re-handling hauls were planned to be performed by the larger trucks.

The Maptek Vulcan™ haulage module was used to calculate the cycle times and distances. Lines were drawn from every bench for each phase to the destinations. Block model blocks were then coded for cycle times and one-way distances reported.

Various haul profiles were developed for different time periods, and haulage cycle times from the pits were estimated for waste and economic material. Base haul cycle times were estimated using the Vulcan™ software, and which were factored for practical operational hauling aspects to reflect realistic cycle times.

Truck spot, load, and dump times were then added to the factored haul cycle times to make up total haul cycle times. Spot and loading times used were taken from the loading unit time estimates.

Table 16-21 shows selected productivity information for hauling units in saprolite at Toroparu.

Table 16-21: Hauling Statistics by Unit Type in Saprolite at Toroparu

Hauling Equipment Type	**Unit**	**Excav1/ ADT**	**Excav2/ Rigid Frame**
Rated Truck Size	t	41-58	91
Truck Fill Factor by Weight	Wet Tonnage Basis %	100%	96%
Typical Total Truck Loading Time * By Excavator	min	3.13	3.34
Total Truck Dumping Time	**min**	**1.0**	**1.1**
Hauling Efficiency Factor	%	Per SRK Estimate	Per SRK Estimate
Production per Unit (100% Available)	t/op hr	Variable	Variable

Source: SRK, 2021. * Includes truck spotting time.

Table 16-22 shows selected information for hauling units in fresh rock at Toroparu.

Table 16-22: Hauling Statistics by Unit Type in Fresh Rock at Toroparu

Hauling Equipment Type	**Unit**	**FEL1/ ADT**	**Excav1/ ADT**	**FEL1/ Scania**	**FEL2/ Rigid Frame**	**Excav2/ Rigid Frame**
Rated Truck Size	t	41-58	41-58	50	91	91
Truck Fill Factor by Weight	Wet Tonnage Basis %	100%	100%	100%	100%	100%
Typical Total Truck Loading Time * By Excavator	min	2.87	3.13	3.51	4.29	2.91
Total Truck Dumping Time	**min**	**1.0**	**1.0**	**1.1**	**1.1**	**1.1**
Hauling Efficiency Factor	%	Per SRK Estimate	Per SRK Estimate	Per SRK Estimate	Per SRK Estimate	Per SRK Estimate
Production per Unit (100% Available)	t/op hr	Variable	Variable	Variable	Variable	Variable

Source: SRK, 2021. * Includes truck spotting time.

Table 16-23 shows selected hauling productivity information for the planned hauling equipment at Toroparu.

Table 16-23: Hauling Productivities by Unit Type at Toroparu

Loading and Hauling Equipment Types	Unit	ADT	Scania Type	Rigid Frame
Production per Unit (100% Available)	t/op hr	Variable	Variable	Variable
Planned Operating Hours per Shift	scheduled op hrs	10.0	10.0	10.0
Planned Operating Hours per Year *	scheduled op hrs	7,100	7,100	7,100
Estimated Mechanical Availability **	%	85%	85%	85%
Actual Operating Hours per Year	op hrs	6,035	6,035	Av. 6,035
Annual Production Capacity per Unit	Mtpa	Variable	Variable	Variable

Source: SRK, 2021. * Includes allowance of 10 days downtime for weather delays. ** Typical mechanical availabilities for trucks used.

Truck hauling productivities were calculated for each type of truck for each year of the mining operations that were used to estimate respective fleet hauling operating hours required, which were then used as a basis for determining the truck fleet requirements. Additional hauling operations by the ADTs included crushed waste backfill to be hauled to the pits for roadway surfacing.

16.3.13 Auxiliary Equipment

Other major mining operations support equipment was previously shown in Table 16-14. The 834K class rubber-tired dozers will primarily perform general dozing and clean-up in areas not worked by the track dozers. A larger 844H class rubber-tired dozer is planned to be added in Year 10, and a second unit in Year 14. The track dozers will be used for drill site preparation, road and ramp development, maintenance of loading areas, waste dumps and stockpiles, and other duties. The graders and water trucks will maintain ramps, haul roads, and operating surfaces. The vibratory compactors will be used in developing new roads or repairing existing roads. The (smaller) excavators will perform site development work including pioneering and drainage diversion ditch development. The major mining equipment fleet size for roads and dumps was based on the general production level, number of active working faces, and allowance for general site conditions (including annual precipitation).

Annual operating hours were estimated for all of the major mining support equipment units, in general, between 2,500 and 3,500 operating hours per unit per year were scheduled for the Toroparu mining operations up to Year 9, and up to 4,400 for Years 10 through 22. Planned support equipment hours were lower for the Sona Hill mining operations, given the lower production amounts.

Mining support equipment includes equipment maintenance units such as fuel/lube trucks, which will deliver to equipment in the field from a central fuel station, heavy duty mechanics' trucks, welders' trucks, tire service truck, forklift, truck crane, and recovery truck.

A low bed transporter (rated for 91t) was included for moving the drills and hydraulic over longer distances around the mine sites. Mine site operations and development will utilize flatbed trucks (with cranes), various movable generator/pumps, light plants (for night shift operations), transport vans, and various service pick-up trucks.

Dewatering will be required for the Main, NW, SE and Sona Hill Pits. At all of the pits a combination of precipitation falling within the outer perimeter of the pit and groundwater inflows into the pit will account for the total volume of water that will need to be handled by the dewatering facilities.

Groundwater inflow will be pumped by submersible type pumps (such as a Flygt BS2290-434, 90 kW, driven by diesel generator), and precipitation inflow by slurry type pumps (such as Godwin HL260M, 560 kW, driven by Caterpillar C-18 diesel engine), which will be capable of handling solids up to 50 mm in diameter. The Godwin pumps will be spaced at maximum vertical stages (intervals) of 90 m in order to pump rainfall from sumps located at the current pit bottom (within the pit phase designs) up to the pit rim.

16.3.14 Mine Labour

The mine department will have salaried staff for mine administration, supervision of mine operations, supervision of mine equipment maintenance, and for technical services (geology and mining departments). Many of these positions will be a permanent day shift. Hourly employees will fill mining production, mining support functions, and mining equipment maintenance positions.

The maximum mine administration and operations supervision staff will total 28 positions up to Year 9 (the end of Sona Hill operations), and up to 25 for most subsequent years. The technical services staff will total 14 positions for most years. Total salaried staff was planned to reach a maximum of 42 positions. Salaried staff requirements were estimated for both expatriate and national employees.

The operations, mine equipment maintenance, and technical services will include:

- Mine administration will include the mine manager and secretary. The mine manager will be an expatriate position up to Year 8;
- Mine operations will include the mine general foreman, shift foremen, drill and blast foreman, mine infrastructure foreman, mine supervisors, dispatch operator, cost controller, and training supervisor;
- Mine maintenance includes the maintenance foreman, senior maintenance supervisors, shift foremen, and supervisors;
- Technical services will include the technical services manager, secretary, and chief surveyor. The technical services manager will oversee the Mining and Geology departments, and will be an expatriate position up to Year 8;
- Mine geology includes the chief geologist, geologist, grade control engineer, and a geotechnical engineer. The mine geologist will handle pit mapping, development drilling, and other resource duties (such as local resource estimation and reconciliations). The grade control engineer will supervise economic material grade control in the mine. The geotechnical engineer will be responsible for monitoring slope stability in the pits and waste dumps, as well as monitoring material compaction and embankment stability at the tailings storage facility. The chief geologist will be an expatriate position up to Year 8; and
- Mine engineering includes the senior mining engineer, short- and long term planners, dispatch engineer (supervising mining equipment deployment), and drafts technicians.

Three mine production and maintenance hourly crews will be necessary (to be rotated on the two-shift system). Equipment operator labour positions were based on the number of mining equipment units required, and on the assumption that some of the operators will be cross-trained. When some of the operators are not required to be on one type of heavy equipment unit, they will be able to operate another. To maintain this, it is planned for the mine department to have an equipment trainer permanently on staff.

Operator positions were estimated for each year of operation. As mentioned, the number of equipment operator labour positions was based on the number of mining equipment units required. Required drilling, loading and hauling fleet equipment numbers were each rounded up to the nearest whole unit required for a year, and each equipment unit was allocated three operators. This operator estimate was adjusted up to allow for a 15% factor for vacation, sickness and absence (VSA). The resulting operator estimate was then adjusted down (by 9%) to target meeting an average 91% mechanical availability (MA) for the mining equipment units.

A mining equipment maintenance department will be staffed with mechanics, electricians, welders and other maintenance personnel. Hourly maintenance man-hours were estimated as 40% of major mining equipment man-hours required.

Mine total hourly labour requirements, including VSA and MA adjustments, are shown in Table 16-24. The hourly labour is divided into mine operations and mine maintenance. The peak number of personnel occurs in Year 10, and the lowest number of personnel occurs in Years 23 and 24 when only stockpile re-handling is taking place.

Annual salaries and annual (hourly paid) wages include burdens for the national staff personnel, and the few expatriate staff planned.

Table 16-24: Mine Hourly Labour Requirements for Selected Years

Mine Hourly Labour	-1	1	2	3	4	6	8	9	10	12	14	16	18	20	22	23	24
Blasting																	
Blaster	3	3	3	3	3	3	3	3	2	2	2	2	2	2	-	-	-
Blasting labourer	12	12	12	12	12	6	12	12	12	12	12	12	12	12	4		-
Subtotal Blasting	**15**	**15**	**15**	**15**	**15**	**8**	**15**	**15**	**14**	**14**	**14**	**14**	**14**	**14**	**4**	**-**	**-**
Equipment Ops																	
Drill operators	6	6	9	9	9	9	9	9	16	13	16	16	16	19	6	-	-
Loading operators	16	19	22	31	28	25	28	28	41	22	25	25	28	25	16	9	3
Truck drivers	35	47	72	94	97	88	104	107	141	113	104	110	100	104	50	13	6
Other mine equipment	53	53	57	57	57	35	57	57	50	50	53	53	53	53	35	22	22
Support equipment	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Subtotal Equip Ops	**113**	**128**	**163**	**194**	**194**	**160**	**201**	**204**	**251**	**201**	**201**	**207**	**200**	**204**	**110**	**47**	**34**
General Mine Ops																	
General mine Ops	21	21	21	21	21	14	21	21	14	14	14	14	14	14	14	7	7
Grade Control Tech	9	9	9	9	9	6	9	9	6	6	6	6	6	6	6	3	3
Surveyor	3	3	3	3	3	2	3	3	2	2	2	2	2	2	2	1	1
Rodman	3	3	3	3	3	2	3	3	2	2	2	2	2	2	2	1	1
Subtotal Eq Ops	**36**	**36**	**36**	**36**	**36**	**24**	**36**	**36**	**24**	**24**	**24**	**24**	**24**	**24**	**24**	**12**	**12**
Total Mine Ops																	
Maintenance																	
Truck fleet mechanics	21	24	30	34	34	28	35	36	42	34	34	35	34	35	20	9	7
Load/support fleet mech	16	18	22	26	26	21	27	27	32	26	26	27	26	26	15	7	5
Field maint mech	16	18	22	26	26	21	27	27	32	26	26	27	26	26	15	6	5
Subtotal Maint	**53**	**60**	**74**	**86**	**86**	**70**	**89**	**90**	**106**	**86**	**86**	**89**	**86**	**87**	**50**	**22**	**17**
Total Hourly	**217**	**239**	**287**	**332**	**332**	**262**	**340**	**345**	**395**	**325**	**325**	**334**	**324**	**329**	**188**	**81**	**63**

Source: SRK, 2019. Including VSA and MA adjustments.

16.4 Underground Evaluation

Underground development will commence at the beginning of the ninth year of open pit operation and targets 3,500 tpd, ramping up to full production over an approximately two-year period. The ramp-up allows for the main ramp system development at the 250 m elevation down from the surface portal in the

NW Pit and to connect to both fresh air and return air raises, providing ventilation and secondary egress for the mine. Underground production is scheduled based on approximately 3,500 tpd mill feed and 750 tpd average waste, excavated using a fleet of 15 and 10 tonne load-haul-dump loaders, hauled with 45 tonne trucks using the ramps to portals entrances and rehandled using the surface fleet. Production is expected to commence in the central area between the Main and NW Pits from 360 Level (approximately 360 m elevation below surface) and continues for the first 2 years in a bottom-up sequence. It is anticipated that mining next transitions to production from lower mining areas below and around Main and NW Pits for approximately the final 10 years of the LoM. Figure 16-13 for the LoM underground design.

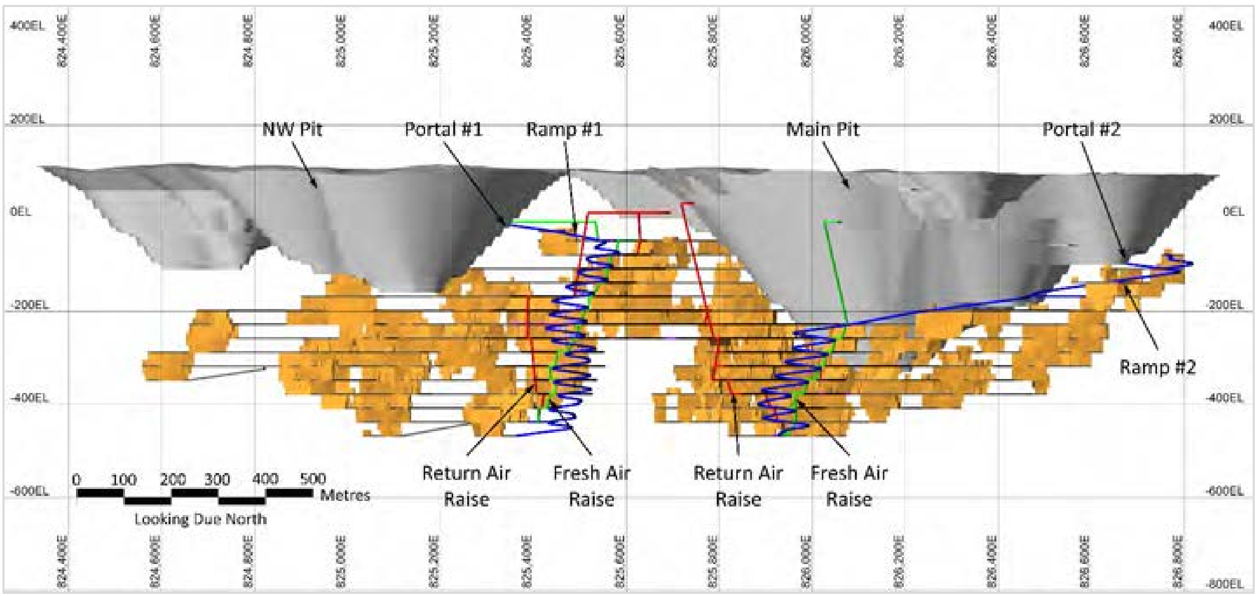


Figure 16-13: Underground Long Section View

16.4.1 Underground Mineral Resources within the PEA Mine Plan Estimate

16.4.1.1 Conversion Assumptions, Parameters and Methods

Conversion assumptions (e.g., mining planned dilution, mining unplanned dilution, mining recovery, process recovery, cut-off grade calculation, MSO, underground development design and costs) were taken into consideration to calculate the mineral resources within the PEA mine plan estimate.

The following steps were used to calculate the mineral resources within the PEA mine plan:

1. Estimate costs, mining dilution, mining recoveries and process recoveries for MSO cut-off grade;
2. Input optimization parameters into Datamine's MSO tool to calculate potential mineable shapes (Measured, Indicated and Inferred resources were included as mineral resources within the PEA mine plan);
3. Removing MSO shapes that are either too isolated or present difficult mining geometry (remaining shapes represent stope material);
4. Complete underground lateral and vertical underground development design to facilitate mining;
5. Sequence and schedule mine plan;
6. Complete detailed mine cost estimates based on detailed mine plan;
7. Prepare a discounted cash flow based on all capital and operating cost inputs; and

8. Select a final mine plan and cash flow followed by reported mineral resources within the PEA mine plan.

16.4.1.2 Underground Mineral Resources within the PEA Mine Plan

The estimate of underground mineral resources within the PEA mine plan is effective as of December 1, 2021 and is presented in Table 16-25. The PEA models an underground mine with mineral resources within the PEA mine plan containing 1.589 Moz of Au, 0.189 Moz of Ag and 29.6 Mlb of Cu (13.4 kt).

Measured, Indicated and Inferred resources were used for conversion to mineral resources within the PEA mine plan within the PEA underground designs. The mineral resources within the PEA mine plan are contained below the Toroparu Pit.

The mineral resources within the PEA mine plan are valid at the time of estimation and include CoG assumptions made before the final PEA cash flow model was completed. Nordmin confirmed the overall project economics are favorable at the approximate four-year moving average Au price of US$1,500/oz Au, an average Ag price of US$20/oz Ag, and an average Cu price of US$3.13/lb Cu.

Table 16-25: Underground Mineral Resources within the PEA Mine Plan

UNDERGROUND MINERAL RESOURCES WITHIN THE PEA MINE PLAN								
Area	**Resource Category**	**Tonnes**	**Au g/t**	**Ag g/t**	**Cu %**	**Contained Au Toz**	**Contained Ag Toz**	**Contained Cu Tonnes**
Main/ NW	Measured	839	2.73	0.63	0.07	74	17	0.6
	Indicated	5,899	3.24	0.49	0.11	614	92	6.2
	Measured & Indicated	6,738	3.17	0.51	0.10	687	110	6.8
	Inferred	7,447	3.77	0.33	0.09	902	80	6.6
	Grand Total	**14,185**	**3.48**	**0.41**	**0.09**	**1,589**	**189**	**13.4**

Source: Nordmin, 2021

16.4.1.3 Relevant Factors

There is no material mining, metallurgical, infrastructure, permitting and other factors that could affect the mineral resources within the PEA mine plan.

16.4.2 Geomechanical

16.4.2.1 General

Knight Piésold provided geomechanical input for conceptual level underground mine design, including rock mass characterization, stope dimensions, stope dilution estimate, crown pillar dimensions, ground support requirements, and general rock mechanics considerations for the Project.

16.4.2.2 Background

The PEA mining strategy developed by Nordmin envisions mining the near surface mineralization at the Main, Southeast and Sona Hill deposits with a series of open pits and extracting the deeper portions of the Main ore body with underground mining. The underground mining strategy includes:

- Mining method – Longitudinal open stoping with a sub-level spacing of 30 m.
- Mining width – Typically between 3 m and 8 m, with a minimum mining width of 2 m and maximum of 16 m.

- Backfill – Cemented paste backfill (CPB).
- Overall access – Ramp access from surface.
- Development Dimensions – The development is planned to be 5 m wide and 5 m high. The span of the overcut and undercuts may be less than 5 m, depending on the width of the ore body.
- Sequence – Underground and open pit mining are planned to be completed concurrently. The underground mining will progress in a series of mining blocks, working in an overhanded sequence. A sill pillar is proposed between the upper and lower portions of the mining blocks to allow the upper part of each block to be mined first. The sequencing of the open pit and the underground mine was not available at the time of the study.
- Interaction with the open pit – The underground mine will be located directly below and behind the walls of the open pits. During the rainy season, water is expected to accumulate at the base of the open pits. This has influenced the design input provided for this study.

The underground mine design input was developed based on an initial mine plan (Figure 16-14) in which the open pits ranged in depth from approximately 100 to 170 metres below ground surface (mbgs) and the underground mine extended from a depth of 75 mbgs to 575 mbgs, with a strike length of approximately 2,500 m. The mine plan was subsequently revised (Figure 16-15), with the open pits increasing in size and extending to a depth of approximately 300 to 470 mbgs. The stopes within the expanded open pit volume were removed but the extents of underground mining were otherwise unchanged. The uppermost stope is now approximately 120 mbgs. As a result, the recommendations provided in this letter are considered to be generally applicable to the revised mine plan.

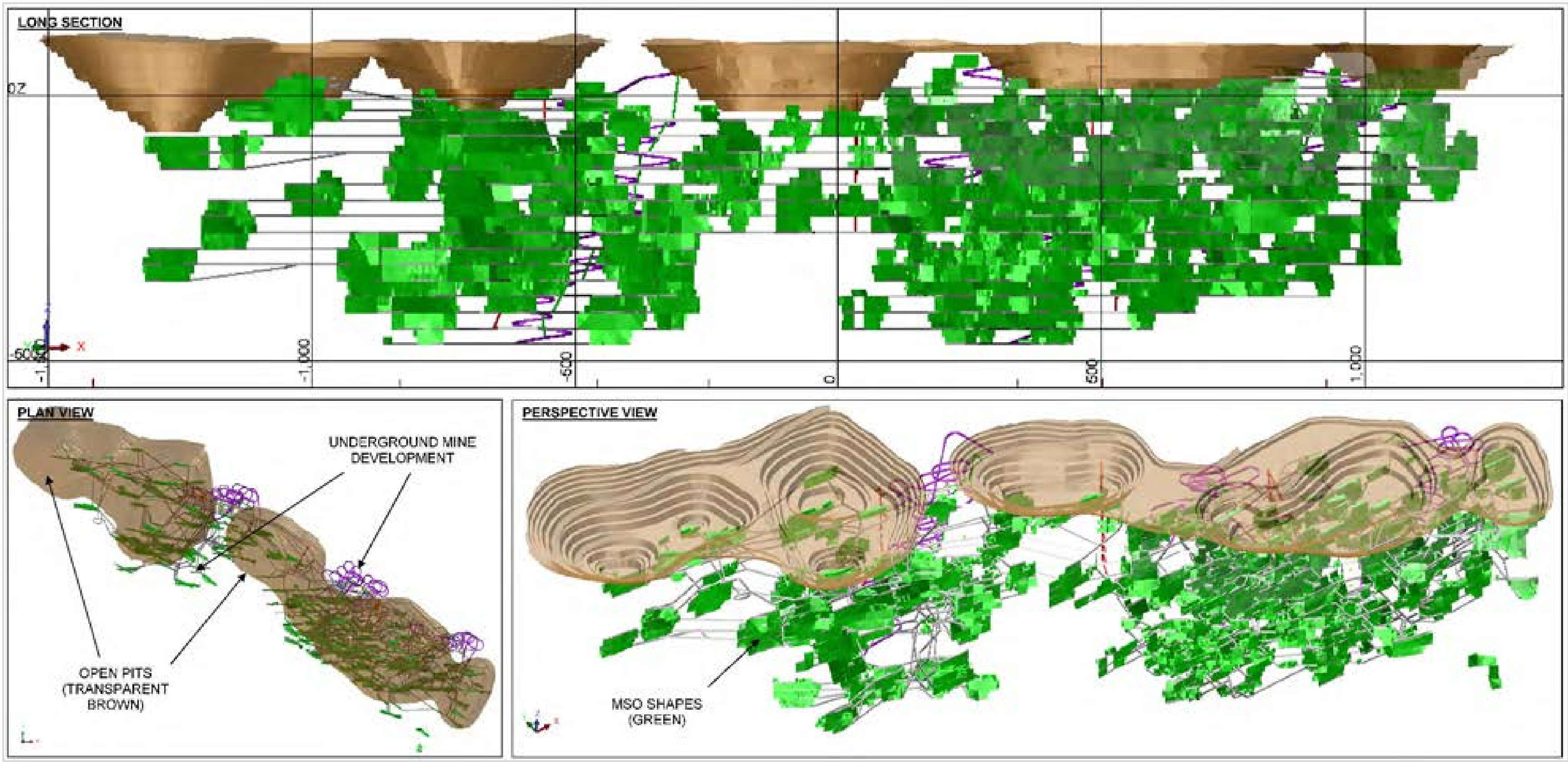


Figure 16-14: Initial mine plan

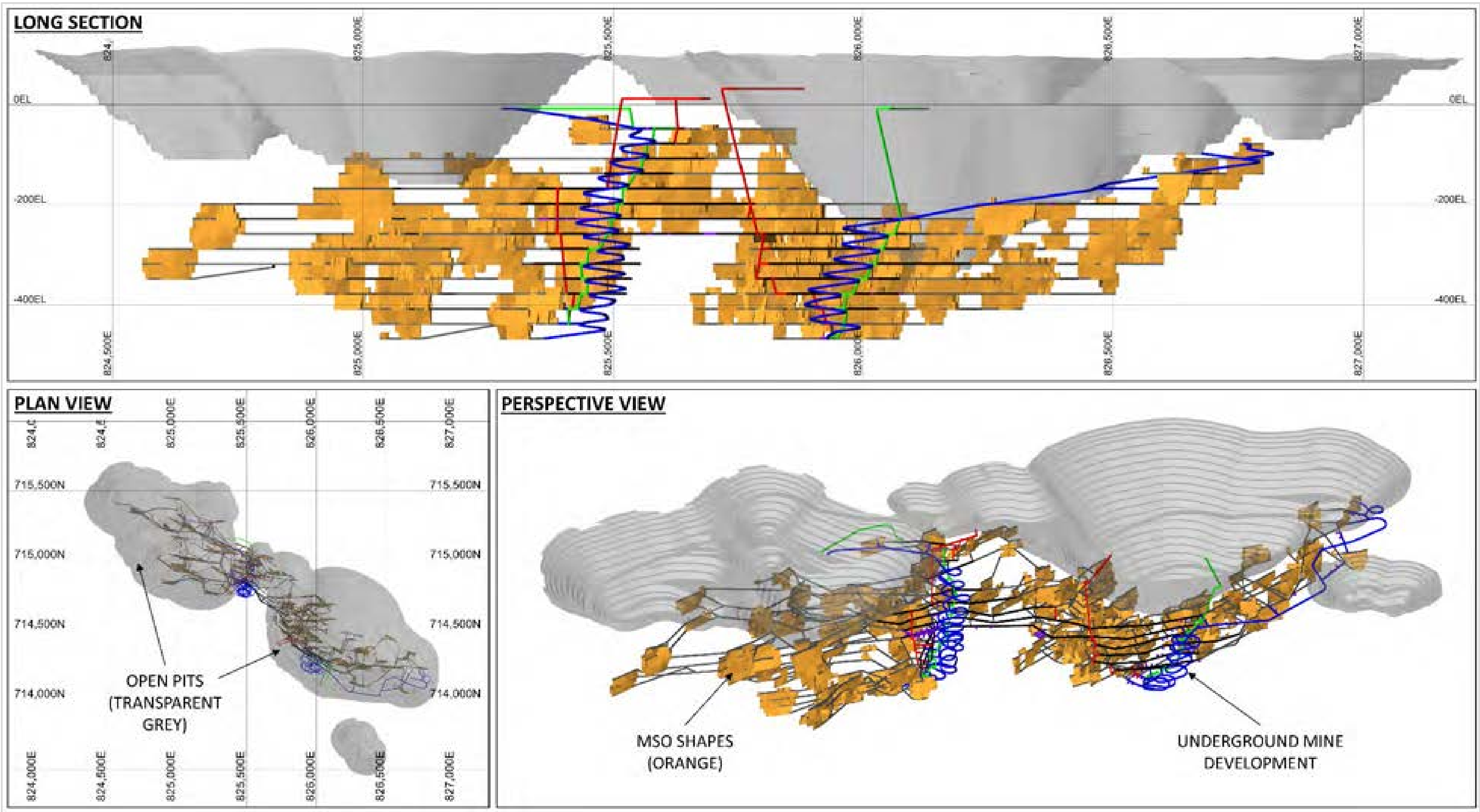


Figure 16-15: Revised mine plan

The geological setting of the Project is described in detail in KP (2018). The Main deposit occurs along a Granodiorite intrusion of a metamorphic volcanic sequence. Numerous volcanic xenoliths are present within the intrusion in the vicinity of the deposits and are referred to as Mixed Facies. The deposit is overlain by Saprolite and a Transition Zone with a combined thickness of approximately 20 m.

16.4.2.3 Rock Mass Characterization

The geomechanical characteristics of the deposit rock masses have been previously described in detail (KP, 2018). A single rock mass structural domain was defined with three joint sets:

- Joint Set 1 strikes NW-SE and dips between 45° and 60° to the SW
- Joint Set 2 is sub-vertical and strikes WNW-ESE
- Joint Set 3 is sub-vertical and strikes NW-SE

Joint Sets 2 and 3 are parallel to the two sets of mineralized structures. The rock mass structure is summarized on Figure 16-16.

The following rock mass quality domains were defined for the Main and Southeast deposits on the basis of lithology and weathering:

- Saprolite
- Intrusives
- Mixed Facies
- Volcanics

The Intrusives, Mixed Facies and Volcanics are typically of GOOD to VERY GOOD quality, with mean RMR_{89} values of approximately 75 (Intrusives) to 80 (Mixed Facies and Volcanics) and a standard deviation of approximately 10. All three domains have mean intact strengths of approximately 130 MPa. The previous characterization focused on the rock masses expected to form the proposed open pit slopes. As a result, the rock masses in the immediate vicinity of the proposed underground openings were reviewed in order to assess whether the previously defined domains and design parameters were applicable to the proposed underground mine design. The review considered the detailed geomechanical logging, drill hole RQD data as well as select core photos provided by the Company. Key conclusions from the review include:

- None of the geomechanical drillholes with detailed geomechanical logging completed to date intersect the proposed stopes.
- The RQD data and core photos from resource drillholes that intersect the proposed stopes suggest that the mineralized zone and immediate hanging wall (HW) and footwall (FW) are of similar quality to the main domains defined in KP (2018).
- The Intrusives domain grouped the Granodiorite with several dykes that intrude the ore body. The review suggests that the Granodiorite is of better quality than the dykes and is best considered as a separate domain for the purposes of the current study. As a result, a separate rock mass quality domain has been defined for the Granodiorite, which is characterized by a mean RMR_{89} value of 80 and a standard deviation of 8. The Granodiorite has a mean intact strength of approximately 120 MPa.
- Localized intervals of FAIR rock mass quality (i.e., RMR_{89} values of 40 to 60) are present within the deposit. While many of these intervals correspond to the modelled faults, other intervals do not.
- The Saprolite is not expected to be encountered in the proposed underground openings as all openings to surface have been planned to daylight within the open pits.

The review is summarized on Figure 16-17 and Figure 16-18. The rock mass properties for the underground design study are summarized in Table 16-26.

Table 16-26: Rock Mass Properties

<table>
<tr><th colspan="2">Rock Mass Properties</th><th>Volcanics</th><th>Intrusives[1]</th><th>Granodiorite</th><th>Mixed Facies</th></tr>
<tr><td colspan="2">Lithological Code</td><td>VOL</td><td>INT</td><td>GRDR and GRDT</td><td>MF</td></tr>
<tr><td colspan="2">Number of Samples</td><td>28</td><td>5</td><td>9</td><td>20</td></tr>
<tr><td>Intact Rock Strength (MPa)</td><td>Mean</td><td>131</td><td>128</td><td>122</td><td>134</td></tr>
<tr><td colspan="2">Number of Runs Measured</td><td>776</td><td>67</td><td>139</td><td>619</td></tr>
<tr><td rowspan="4">RMR_{89}</td><td>Median</td><td>80</td><td>77</td><td>80</td><td>80</td></tr>
<tr><td>Mean</td><td>79</td><td>77</td><td>82</td><td>80</td></tr>
<tr><td>Std. Dev.</td><td>8</td><td>10</td><td>8</td><td>9</td></tr>
<tr><td>Description</td><td>GOOD</td><td>GOOD</td><td>VERY GOOD</td><td>GOOD</td></tr>
</table>

Notes: 1. The intrusive unit excludes the granodiorite units.

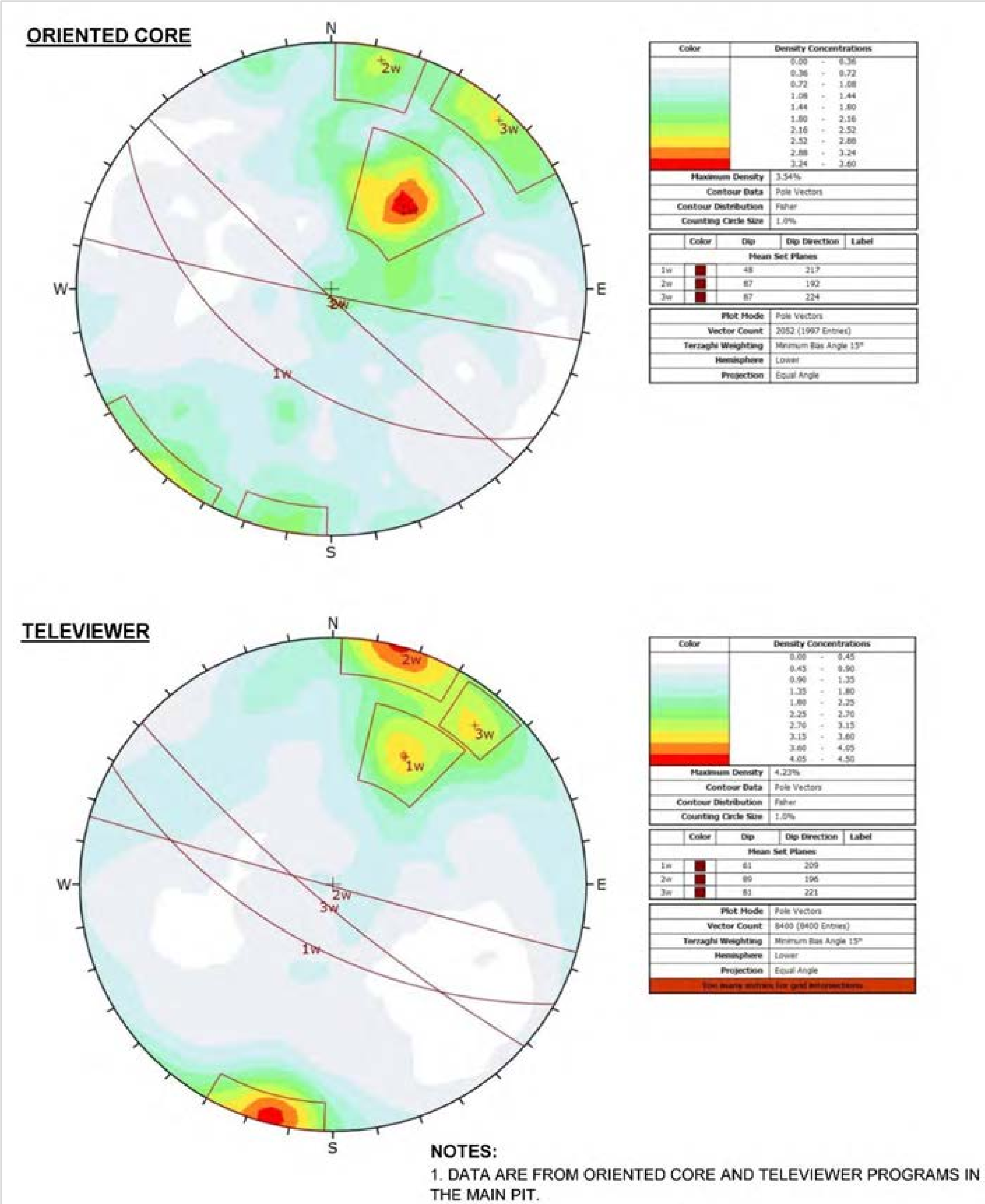


Figure 16-16: Stereonet plot of joints and foliation

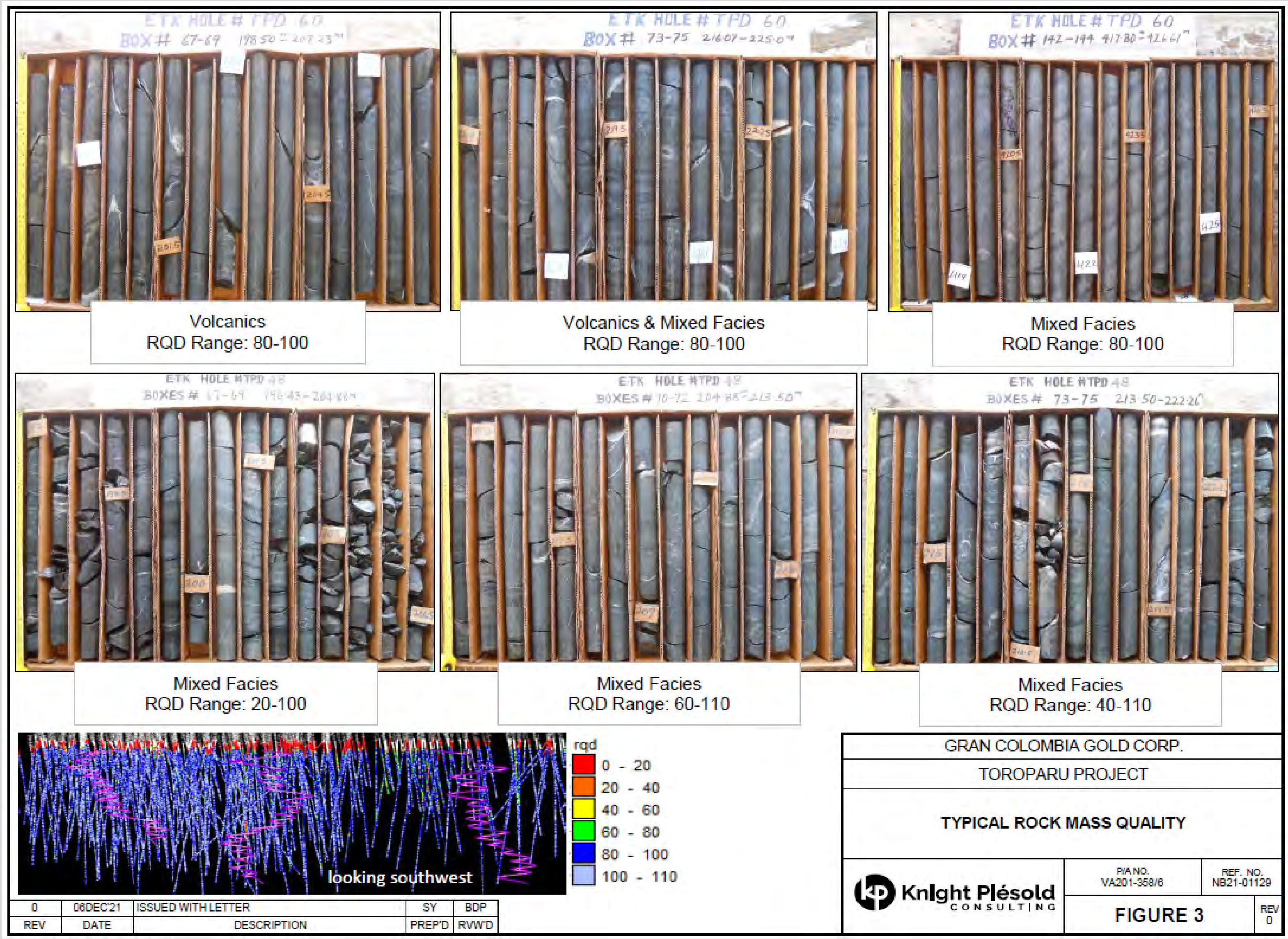


Figure 16-17: Typical rock mass quality

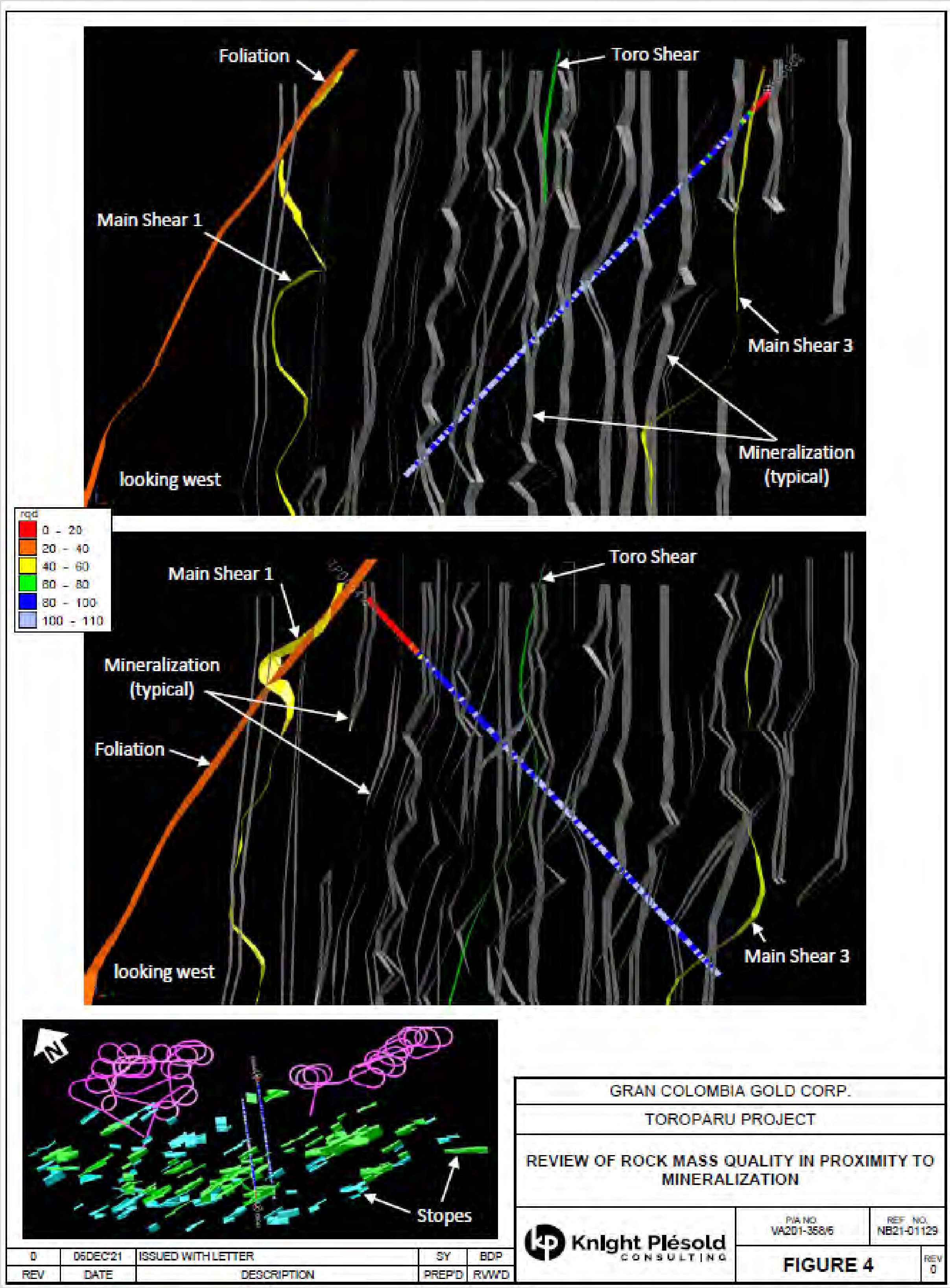


Figure 16-18: Review of rock mass quality in proximity to mineralization

16.4.2.4 Underground Mine Design Input

16.4.2.4.1 General

Stability analyses have been completed in order to provide PEA level underground rock mechanics design input for longitudinal open stoping of the Main and Southeast deposits at the Project. Guidance was provided on the following:

- Stope dimensions and estimated stope dilution
- Backfill strength
- Crown pillar dimensions
- Ground support

The completed stability analyses and subsequent recommendations are described in the following sections.

16.4.2.4.2 Stope Dimensions and Dilution

Achievable stope dimensions were assessed using the Stability Graph method (Potvin, 1988 and Nickson, 1992). This method focuses on assessing the stability of the walls and back of the stopes based on the initial excavation geometry and expected rock mass characteristics. The expected dilution for the recommended stope geometries was estimated using the Equivalent Linear Overbreak/Slough (ELOS) Method (Clark, 1998), which relies on the same inputs as the Stability Graph method. ELOS is an estimate of the average thickness of dilution over the stope surface. The inputs to the analyses are summarized below.

- Rock Mass Quality – The analyses considered the Granodiorite, Volcanics and Mixed Facies domains. In order to consider the potential effects of zones of reduced rock mass quality within the deposit, a RMR_{89} value of 60 was also considered as a sensitivity case.
- Stope Height – 35 m (sub-level spacing of 30 m with a 5 m high overcut).
- Stope Dip – 70° to 90°.
- HW-FW Span – 8 m to 12 m. Stopes with a HW-FW width exceeding 12 m will be panelled.

The analyses suggest that a 25 m strike length is achievable under the expected typical conditions (i.e., rock mass qualities within one standard deviation of the mean RMR_{89} value). The associated dilution is estimated to be approximately 0.5 m for the HW and 0.5 m for the FW. The analyses assume that long support is installed in the back of stopes with a span of 10 m or greater. The dilution is predicted to increase to between 1 m and 2 m within zones of reduced rock mass quality (i.e., RMR_{89} values of 60).

The following should be considered when relying on the stope dimension recommendations and dilution estimates:

- The Stability Graph and ELOS methods focus on the rock mechanics considerations influencing dilution. Operational considerations such as stand-up time and drilling and blasting practices will have a significant impact on the amount of dilution. The recommendations assume that the stopes will be mined and backfilled in a timely way.
- Longitudinal mining provides an opportunity to control HW performance and dilution by adjusting the strike length of the stopes on a case-by-case basis.
- Experience at other operations indicates that undercutting the HW typically results in increased dilution. Care should be taken when considering stopes with a HW to FW span less than the span of the undercut.

- Increased stope dilution is possible where faults are intersected.

16.4.2.4.3 Backfill Strength

Strength requirements for the paste backfill were estimated using the empirical methods developed by Mitchell et al. (1982) and Mitchell (1991). The analyses considered a stope with a 25 m strike length and 8 m to 12 m HW-FW width backfilled to a height of 30 m. The paste backfill was assumed to have a unit weight of 20 kN/m^3 and an internal angle of friction of 30°. The three scenarios considered, and the associated backfill strength requirements, are summarized below.

- Exposing the end wall of the stope when retreating: 300 kPa
- Exposing the HW or FW of the stope when panelling a wider part of the ore body: 400 kPa
- Undercutting a stope: 1, 1.5 and 2 MPa for spans of 8, 10 and 12 m, respectively

Note that the strength requirements for undercutting a stope assume that man-entry below the paste backfill is limited to an overcut with a span of 5 m.

16.4.2.4.4 Crown Pillar Dimensions

The mine plan incorporates a crown pillar between the open pits and the proposed underground workings. The stability and required dimensions of the crown pillar were assessed using the Critical Scaled Span method (Carter, 1995). Note that the analyses did not consider the potential impacts of underground mining on the stability of the open pit slopes. Due to the climate, two cases were evaluated: one during the dry season where the open pit is dry, and one during the rainy season where water will be stored in the open pit. The presence of this water during the rainy season significantly increases the potential consequences of a crown pillar failure and this is reflected in the analyses. The inputs to the analyses are summarized below.

- Rock Mass Quality – The analyses considered the Granodiorite, Volcanics and Mixed Facies domains.
- Groundwater (Jw) – A Jw of 1 (dry or minor inflow) was used for the dry season. A Jw of 0.66 was used to account for increased inflow and porewater pressure within the crown pillar due to the water in the open pit during the rainy season.
- HW-FW Span – The span was varied between 8 m and 10 m.
- Target Probability of Failure (PoF) - The following acceptance criterion based on Carter (1995) were used to account for the change in climate throughout the year:
    - A PoF of 5% was selected for the dry season as it is applicable to short to medium term/semi-temporary to semi-permanent openings.
    - A PoF of 1.5% was selected for the rainy season to reflect the increased consequences of a failure associated with the water stored in the open pit above the crown pillar.

The results of the assessment suggest that:

- The stability of the crown pillar is sensitive to the HW-FW span of the underlying stopes. A maximum HW-FW span of 8 m is recommended for the stopes directly below the crown pillar.
- A crown pillar thickness of 25 m (i.e., one sub-level, less the height of the overcut) is required to meet the target PoF of 5% below a dry open pit or surface. In a few instances the calculated PoF exceeded the target by less than 1%. These cases can be managed with instrumentation, inspections, and ground support. A similar offset is recommended for stopes behind the final walls of the open pit that are above the possible pit lake elevation.

- A crown pillar thickness of 50 m is required to meet the target PoF of 1.5% below a partially flooded open pit. A similar offset is recommended for stopes behind the final walls of the open pit that are below the possible pit lake elevation.

There is the potential to recover at least part of the crown pillars below the open pits during the dry season, subject to the following controls:

- Mining will only be undertaken within the crown pillar during the dry season.
- The stopes mined within the crown pillar, including any overcuts, will be tight-filled with paste backfill to provide long term support for the crown pillar and to limit any waterflow into the underground mine openings during the wet season.
- Plugs or bulkheads will be installed at the level access to any stopes mined within the crown pillar prior to the start of the next wet season. The intent is to isolate the mining within the crown pillar from the rest of the mine in the event of an inrush. If re-entry to an area is required during a subsequent dry season, there will need to be a method to check for and drain any water that may have accumulated behind the plug or bulkhead.
- Instrumentation will be installed in the permanent crown to monitor pillar performance (e.g., in the intersections between the ore drive and the level accesses).
- A detailed technical study is completed on the stability of the crown pillars, adjacent open pit slopes, and the potential for an inrush event.

16.4.2.4.5 Ground Support

Guidance on ground support requirements was provided to support the estimation of mining costs. The recommendations were developed using empirical methods, Unwedge (Rocscience, 2019) and typical practice in the Canadian mining industry. The analyses considered development openings with a span of 5 m and intersections with a span of 7.5 m. The need for ground support in the back of the stopes was also considered. The recommended ground support is summarized below.

- Development – 2.4 m long #6 resin rebar in the back and 1.8 m long #6 resin rebar in the walls installed on a 1.2 m square pattern with 6-gauge galvanized welded wire mesh. The mesh and bolts should extend to within 1.5 m of the floor. Galvanized friction sets can be used in place of rebar in the walls in short-term development (e.g., overcuts).
- Intersections – 3.6 m long spin cables on a 1.8 m square spacing are recommended in addition to the primary development ground support. 0-gauge mesh straps should be installed on pillar corners. The use of straps is most important for acute-angled pillars as well as pillars in adverse conditions (e.g., reduced rock mass quality or adverse structure). For costing purposes, it can be assumed that 0-gauge straps will be required on 30% of the pillar corners.
- Stopes – The stope design assumes the installation of long support in the back of stopes with a HW-FW span of 10 m or greater. Four rows of 5 m long twin cement-grouted cables are recommended in the back of the overcut of stopes. The analyses identified the potential for a wedge to overtop the cables in the stopes oriented NW-SE, though the analyses are sensitive to the position of the wedge and the persistence of the discontinuities forming the wedge. Three rows of three cables or Super Swellex (Pm24) are recommended in the brow of stopes with a HW-FW span of 6 m or greater.

Upgraded ground support, such as shotcrete or cable bolts, is expected to be required when adverse conditions are encountered (e.g., faults, adverse structure, etc.). Shotcrete will be required in the back and shoulders of the overcuts established below paste backfill.

16.4.2.5 Recommendations for Future Work

Future work should include more detailed analyses based on additional geomechanical data for the deposit in order to support the next level of design. Additional data requirements include:

- Additional geomechanical diamond drillholes with core orientation and/or televiewer surveys to confirm the rock mass characteristics in the vicinity of the proposed underground openings, particularly near the transition with the open pits and in the northwest extension of the Main deposit.
- Additional laboratory strength testing in each of the major lithology units to better define the intact rock properties.

Any further infill or definition drilling should be reviewed for opportunities to collect additional information on the deposit rock masses and large scale structures. For the next level of design, the domain definition, stability analyses and slope recommendations should be updated to account for the results of the additional site investigations and any changes to the geological models, large scale structural interpretations and/or underground geometry. Additional domain definition and stability analyses should be completed including:

- The spatial variation in rock mass quality within the deposit should be further investigated.
- The extraction sequence and the stability of the pillars formed between stopes extracting the two different sets of mineralized structures should be reviewed.
- A 3D numerical model should be developed to evaluate the interaction between the open pit and underground mine. The results of the model can also be used to comment on infrastructure placement, extraction sequencing and pillar stability.
- The stability of the open pit slopes should be re-evaluated to account for the underground openings planned below and behind the open pit walls.
- The requirements for ground support in the stopes should be further evaluated and refined.
- The position of the proposed portals within the open pits should be reviewed and ground support requirements evaluated.

16.4.3 Mining Method Selection

The mining methods for the Project were selected based on economic and geotechnical parameters, ensuring it was suitable for the mineralization geometry. Bulk underground mining methods, such as block caving, were deemed impractical due to the geometry of the deposit. Transverse longhole open stoping was explored but rejected due to the discontinuous and relatively thin nature of the mineralization. Drift and fill was explored but rejected due to insufficient average grade of the mineralization.

The longitudinal retreat LHOS mining method was ultimately selected. LHOS requires drifting along mineralization via overcut and undercuts and is well suited to following the dual mineralized and thin structures that exist within the deposit. LHOS provides good mining recovery and ensures that stopes are backfilled with some form of cemented backfill, which is essential for concurrent excavation of the open pit.

16.4.4 Cut-off Grades

The mining cut-off for the MSO underground inventory was generated based on a 2.0 g/t gold cut-off grade (insitu), which approximately equates to a 1.6 g/t gold mill feed grade. Table 16-27 shows the cut-off grade parameters used to generate the MSO underground inventory.

Table 16-27: MSO Cut-off Grade Parameters

Parameter	Value
Currency Used for Evaluation	US$
Underground Mining Cost includes assumptions for operating labour, consumables, power, surface hauling	25.77$/t processed
Underground Support Cost Includes assumptions for infill diamond drilling, equipment maintenance, technical services	10.40$/t processed
Process Cost includes assumptions for milling, tailings, water treatment	15.50$/ t processed
G&A Cost Includes assumptions for camp, off site materials transportation	5.95$/t processed
Operating Cost Marginal Allowance 10% Marginal allowance to cover a portion of capital costs	5.80$/t processed
Selling Cost Includes doré transportation, refining, and 8% government royalty	123$/t.oz.
% Payable	99.95%
Metal Price	1,500 US$/t.oz.
Mining Dilution	15%
Mining Recovery	95%
Process Recovery	87.85%
Production Rate Assumption	3,500 tonne per day

A 1.25 g/t incremental mill feed cut-off grade was selected to apply to development. The amount of incremental development material between 1.25 g/t and 1.6 g/t is 1.6% of total mill feed and 12.8% of total development.

16.4.5 Stope Optimization, Dilution and Recovery

The underground mineralization was evaluated using Datamine's MSO tool to create the mineable inventory. The MSO tool evaluates the deposit based on economic and geometric parameters. The economic parameter used for evaluation was a 2.0 g/t gold grade cut-off, as described in Section 16.4.4. The block model was created in Datamine using 2 m x 2 m x 2 m parent blocks and allowing sub-cells to a minimum of 0.5 m x 0.5 m in the X and Y direction, and variable in the Z direction.

As discussed in Section 14.2, the mineralization within the Main and NW mining areas is contained within mineralized structures (E-W and NW-SE). Preliminary attempts to model the mineable inventory by evaluating the mineralized structures together, resulted in a poor approximation of the NW-SE structure due to its rotated geometry. To better approximate the mineable inventory, the two mineralized structures were separated into unique block models. Each block model was evaluated separately via the MSO tool, using geometric frameworks appropriate for each mineralized structure orientation. This generated two sets of overlapping mineable inventories. The two sets of mineable inventories were merged into one, using Boolean operation tools within Deswik. This merged singular model was used as the basis for the mineable inventory for the Project.

The MSO tool can create full-sized stope shapes, as well as sub-stope shapes. Sub-stope shapes were designed to be half the height and one third the length of full-sized stope shapes. Sub-stope shapes were included in the mineable inventory under certain situations; 1) the sub-stope shape(s) were directly adjacent to a full-sized stope shape and were assumed to be mined with the full-sized stope or 2) the sub-stope shapes were directly above a sill drift and were assumed to be mined via uphole stoping. Table 16-28 shows the MSO parameters used to generate the mineable inventory.

Table 16-28: MSO Parameters

Parameter	Value
Height	30 m, 2 Sub-shapes possible @15 m
Length	25 m, 4 Sub-shapes possible @6.25 m
Width	Variable 2 m to 25 m
Minimum Dip	50 °

The primary sources of external dilution were assumed to be generated from the HW and FW. Lesser amounts of dilution were assumed from the retreat stope sidewalls via paste backfill sloughing and the undercut floor via unintentional downwards mucking. External dilution from the HW and FW will be primarily managed by calibrating drilling and blasting practices to mine wide empirical data collected during the first year of underground production. Mining recovery was determined based on the mining method and similar mining projects. A mining recovery of 92.5% was applied to downhole stopes and 80% for uphole and crown pillar stopes. Table 16-29 shows the dilution and recovery factors applied to stopes.

Table 16-29: Dilution and Recovery Factors

Factor	Value
Mining Dilution – Stoping	12%
Mining Dilution – Development	5%
Mining Recovery – Stoping (Excluding Uphole & Pillar)	92.5%
Mining Recovery – Uphole Stoping	80%
Mining Recovery – Pillar Stoping	80%
Mining Recovery – Development	5%

16.4.6 Stope Design

Stopes are accessed by overcut and undercut stope access drifts which extend from the level haulages. The majority of stopes are assumed to be downhole stopes, however, as mentioned in Section 16.4.5, some sub-stope shapes are designed to be mined via uphole stoping.

As outlined in Section 16.4.1, maximum stope size was governed by geomechanical considerations. Stopes are designed on a maximum 30 m level spacing, 25 m length and 12 m HW – FW span. Stopes shapes created via the MSO tool that have a HW – FW span of greater than 12 m, are assumed to be panelled mined, effectively decreasing the HW – FW span by half.

Downhole stopes are drilled, blasted, and backfilled via overcut access. Stopes with a HW – FW span of less than 5 m are designed to be drilled with a top-hammer production drill and stopes with a HW – FW span of greater than 5 m are designed to be drilled with an in-the-hole (ITH) production drill. Stopes are blasted by first blasting a drop raise to create void, and then blasting the remainder of the rings once sufficient void has been created. Once a stope is mucked out via the undercut access, a fill barricade is constructed, and the stope backfilled with cemented paste backfill (CBP). CBP is delivered via a series of boreholes and pipes from the surface CBP plant.

Upholes stopes are drilled, blasted, mucked, and backfilled via undercut access. Upholes stopes are designed to be drilled with a top-hammer production drill. Stopes are blasted by first blasting an inverse drop raise to create void, and then blasting the remainder of the rings once sufficient void has been created. Once a stope is mucked out, a fill barricade is constructed in the undercut access and the stope backfilled with cemented paste backfill (CBP). CBP is delivered via a series of boreholes and pipes from the surface CBP plant.

CBP binder content required for backfilling stopes varies depending on the stoping requirements for adjacent stopes. Longitudinal retreat LHOS design requires that the end wall of an adjacent mined stope does not collapse into the open void created by mining its neighbour. Cement binder is added to backfill to provide internal cohesion and prevent the column of backfill to collapse when a void is adjacently opened. As a general rule, the larger the exposed column of backfill is (e.g., HW – FW span), the higher the percentage of cement within the backfill is required. Table 16-30 shows the required binder content for various types of stopes.

Table 16-30: Stope Binder Percentages

Stope Scenario	Binder Percent (%)
Final stope in retreating sequence or isolated stope	2%
Stope with width up to 10 m	3%
Stope with width between 10 m and 12 m	4%
Greater than 12 m width, mined in two panels	5% (first panel 6%, second panel 4%)

16.4.7 Mine Access and Development Design

The underground is accessed via two portals from the open pits. The first portal is excavated from the NW Open Pit in year 9 and the second portal is excavated from the Main Open Pit in year 12.

The portals connect to decline ramp systems that extend to a depth of 570 metres. Level accesses are driven from the declines, on 30 metre spacing, and connect to level haulages. Level haulages traverse adjacent to the ore body, offset 35 metres or more from stopes. A series of stope access drifts are driven from level haulages to access stopes. Table 16-31 outlines the sizing for each lateral development type and Figure 16-16 shows a typical lateral development level design.

Table 16-31: Lateral Development Size

Development Type	Size
Main ramp, level access, level haulage, infrastructure	5.0 m wide x 5.0 m high
Stope access to multiple or wide stopes	4.5 m wide x 4.5 m high
Stope access to isolated or narrow stopes	4.0 m wide x 4.0 m high

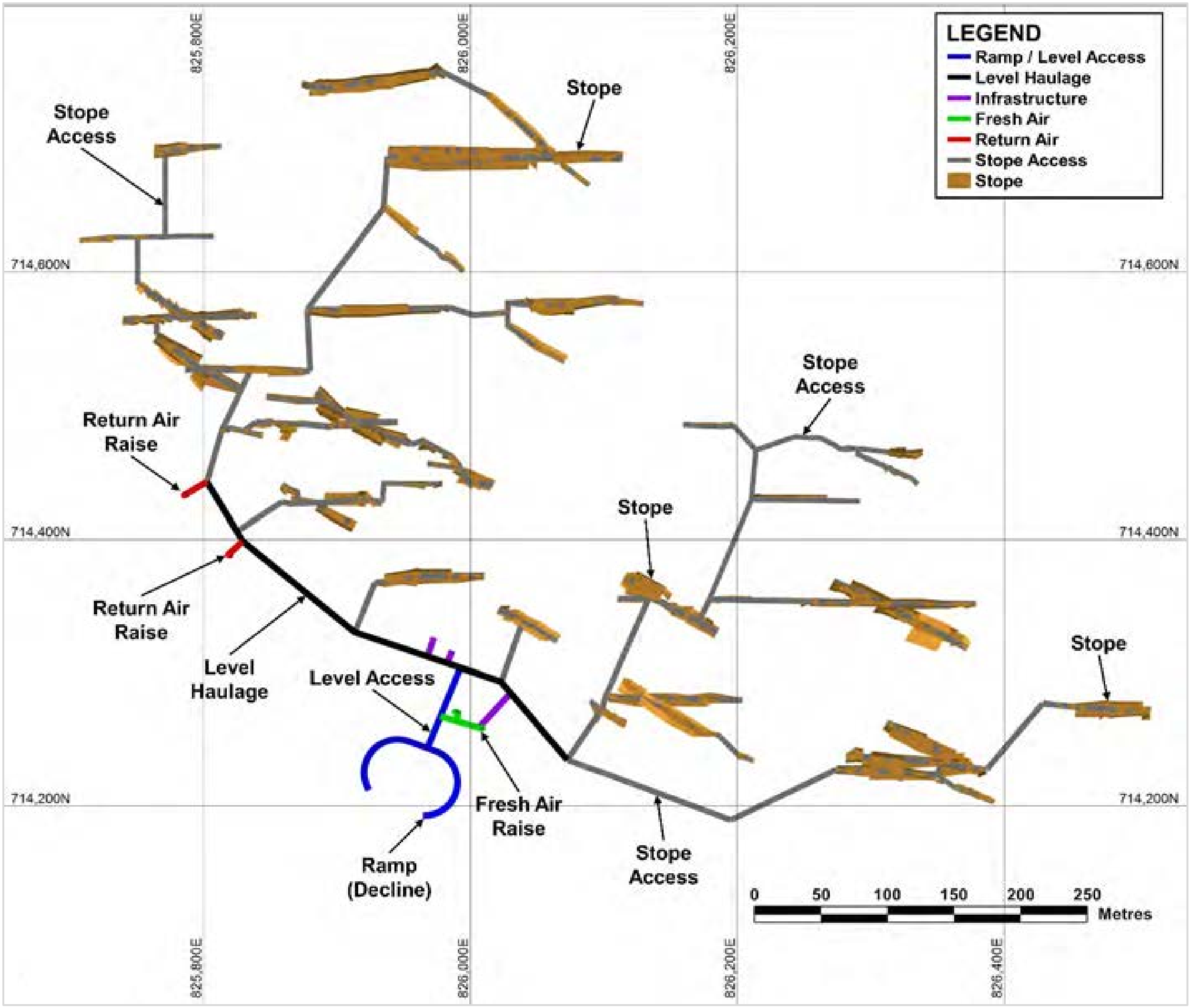


Figure 16-19: Typical lateral development level design

16.4.8 Production

Mill feed is primarily sourced from stope production, however 12.5% of mill feed is expected from lateral overcut and undercut stope access drifts.

Stopes are drilled via longhole drill and loaded / blasted via hand (downholes) or emulsion loader (upholes). Stopes are excavated via 10 tonne and 15 tonne LHDs and material is deposit into remucks close to the level haulages. 15 tonne LHDs load the material from the remucks into 45 tonne haul trucks. The haul trucks transport material up the ramp system and deposit material outside the portal accesses. This material is rehandled by a loader into 50 tonne surface trucks and hauled to the mill. Waste material is handled similarly, however ultimately transported by 50 tonne surface trucks to the waste dump.

Table 16-32 shows the (LHD) mucking cycle times by mining area and Table 16-33 shows the underground haul truck cycle times by mining area.

Table 16-32: Mucking Cycle Times

Mining Block	Round Trip Time (Minutes)
100	3.9
200	5.2
300	5.6
400	4.5
500	7.2
600	5.2
700	8.6
800	7.2
900	11.1
1000	6.1

Table 16-33: Haulage Cycle Times

Mining Block	Round Trip Time (Minutes)
100	38.4
200	45.0
300	38.9
400	57.9
500	54.9
600	41.7
700	57.9
800	61.2
900	54.8
1000	58.1

16.4.9 Productivity

Productivities were developed from first principles. Rates were adjusted based on benchmarking the experience. Table 16-34 shows the shift productivity rates followed by a description of the general and activity-specific parameters upon which the productivity rates are based.

Table 16-34: Shift Productivity Rates

Shift Productivity Rate:	Value	Unit
Annual Mining Days	360	Days/year
Mining Days per Week	7.0	Days/week
Shifts per Day	2.0	Shifts/day
Shift Length	11.5	Hours/shift
Shift Change (Beginning and End)	1.15	Hours/shift
Lunch Time (inc. travel to lunchroom)	0.65	Hours/shift
Equipment Inspection/Refuel	0.25	Hours/shift
Total Available Work Time	9.5	Hours/shift
Worker Utilization	50.0	Min/hour
Effective Work Time per Shift	7.9	Hours/shift

16.4.10 Sequencing

Due to the discontinuous and widespread nature of the mineable inventory, stopes were organized into mining groups to assist sequencing. Ultimately, the mineable inventory was grouped into 92 unique mining groups. Each mining group is designed to be mined independently of other mining groups; therefore, each mining group can be sequenced independently of one another. Figure 16-20 shows the 92 unique mining groups.

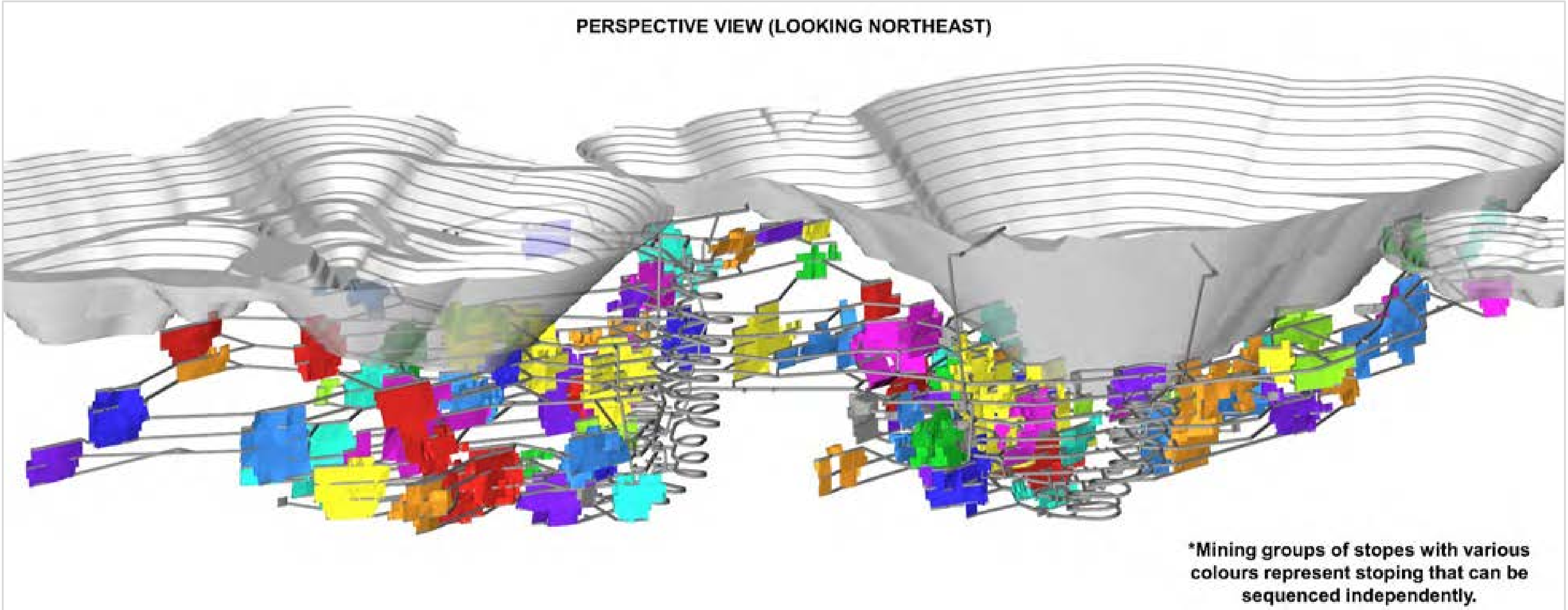


Figure 16-20: Underground mining groups

Stopes within each mining group are sequenced in a conventional bottom-up longitudinal retreat toward the level accesses. Figure 16-21 shows a typical bottom-up longitudinal retreat sequence

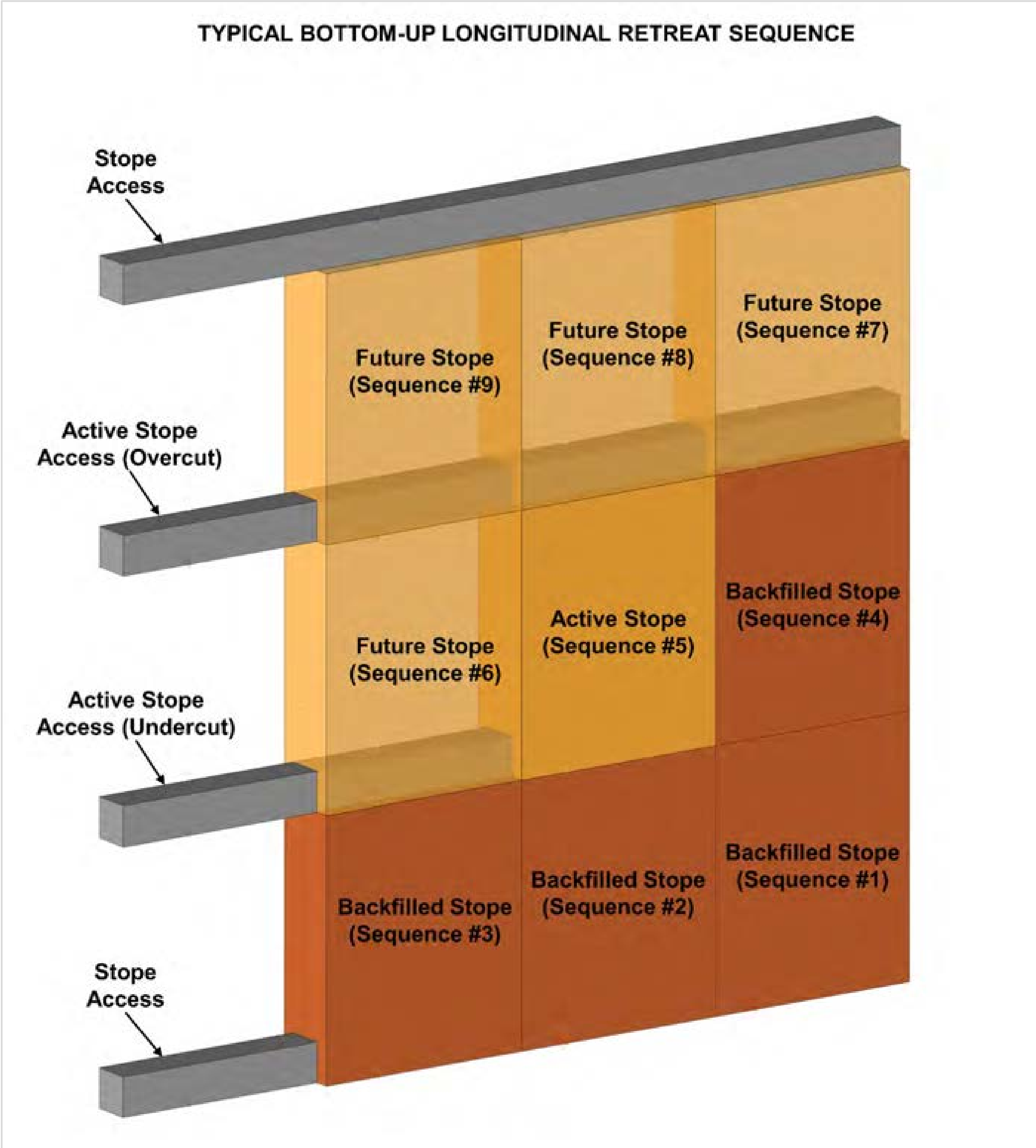


Figure 16-21: Typical bottom-up longitudinal retreat sequence

16.4.11 Development and Production Schedule

The development and production schedule was created based on the input parameters from the mine design created in Deswik and Excel spreadsheets. The average stope cycle time is based on drilling, blasting, mucking, truck hauling and backfilling activities.

Table 16-35 shows average LHOS cycle time.

Table 16-35: LHOS Average Stope Cycle Time

Activity	Days
Drilling	10.6
Blasting	4.5
Mucking	8.1
Backfill Preparation	3.0
Backfilling	1.7
Backfill Curing	14.0
Total Days	**41.9**

The mining operation schedule is based on operating 360 days per year, 7 days per week, with two 11.5 hour shifts per day. A production rate of 3,500 tpd was targeted, with a 36-month ramp-up period to full production. The timeframe of the production schedule is yearly. Table 16-36 shows the development crew requirements for each year.

Table 16-36: Development Crew Requirements

Schedule Year	Development Crews Required
Year 9	1 crew (3 months @ 2 crews required)
Year 10	2 crews
Year 11	4 crews
Year 12	3 crews
Year 13	3 crews
Years 14 – 21	2 crews
Year 22	-

Underground development begins at the start of year 9 with the excavation of portal #1 in the NE wall of the NW Pit. The development of decline ramp #1 (originating from portal #1) continues throughout years 9 and 10, and is completed at the end of year 11, upon reaching 570L.

In year 10, development priority transitions from ramp development to haulage and stope access development. In late year 10, production ramp-up begins in Block 200, and is followed by Blocks 300 and 400 in year 11. In year 12, full production is achieved with the addition of Block 500 production.

Concurrently, in year 12, Portal #2 is excavated in the east wall of the Main Pit and development of the ramp #2 decline begins. Development of ramp #2 continues for the entirety of year 12, reaching its final depth of 570L at the end of year 13.

Production continues in Blocks 400 and 500 throughout years 12 and 13. In year 14, production in Blocks 600, 700 and 800 begins. Production of Blocks 700 and 800 continues until year 18, when production of Blocks 900 and 1000 begins, until end of mine life in year 22. Table 16-37 outlines development metres by block, Table 16-38 outlines the underground production schedule by block, and Table 16-39 outlines the mill feed by year. Figure 16-22 shows the underground mine design by block and Figure 16-23 shows the underground production schedule by year.

Table 16-37: Underground Development Schedule by Mining Block (m)

Underground Develop. (m)	Schedule Year 9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Block 100	3,727	636	-	-	-	-	-	-	-	-	-	-	-	-	4,363
Block 200	-	4,854	-	-	-	-	-	-	-	-	-	-	-	-	4,854
Block 300	-	480	7,042	-	-	-	-	-	-	-	-	-	-	-	7,522
Block 400	-	-	4,125	1,033	1,455	384	-	-	-	-	-	-	-	-	6,998
Block 500	-	-	2,306	6,698	2,749	828	-	-	-	-	-	-	-	-	12,581
Block 600	-	-	-	1,765	1,340	282	810	-	-	-	-	-	-	-	4,197
Block 700	-	-	-	-	2,197	3,296	2,868	2,373	2,872	-	-	-	-	-	13,605
Block 800	-	-	-	979	2,493	2,590	2,297	2,915	3,555	-	-	-	-	-	14,828
Block 900	-	-	-	-	-	-	-	-	609	3,824	2,482	3,376	4,007	-	14,298
Block 1000	-	-	-	-	-	-	-	-	-	2,559	2,404	1,276	618	-	6,857
Total Lateral Development	**3,727**	**5,971**	**13,473**	**10,475**	**10,235**	**7,379**	**5,974**	**5,288**	**7,036**	**6,383**	**4,885**	**4,651**	**4,626**	**-**	**90,104**
Total Vertical Development	**378**	**140**	**1,114**	**233**	**91**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**1,956**

Table 16-38: Underground Production Schedule by Mining Block (kt)

Underground Production (kt)	Schedule Year 9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Block 100	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Block 200	-	251	196	-	-	-	-	-	-	-	-	-	-	-	446
Block 300	-	-	460	335	-	-	-	-	-	-	-	-	-	-	794
Block 400	-	-	88	190	359	276	60	-	-	-	-	-	-	-	973
Block 500	-	-	-	736	889	429	108	-	-	-	-	-	-	-	2,162
Block 600	-	-	-	-	11	31	120	82	-	-	-	-	-	-	244
Block 700	-	-	-	-	-	301	587	738	688	236	-	-	-	-	2,550
Block 800	-	-	-	-	-	226	386	440	566	303	-	-	-	-	1921
Block 900	-	-	-	-	-	-	-	-	5	331	455	569	891	484	2,735
Block 1000	-	-	-	-	-	-	-	-	-	390	806	692	354	118	2,359
Total Mineralized Material	**-**	**251**	**744**	**1,260**	**1,260**	**1,262**	**1,261**	**1,260**	**1,260**	**1,260**	**1,261**	**1,261**	**1245**	**602**	**14,185**
Waste Material	285	310	769	405	454	183	147	132	219	186	139	126	134	-	3,489
Total Material	**285**	**561**	**1,513**	**1,666**	**1,714**	**1,445**	**1,407**	**1,392**	**1,479**	**1,446**	**1,400**	**1,386**	**1,379**	**602**	**17,674**

Table 16-39: Underground Mill Feed by Year (kt)

Underground Mill Feed	Schedule Year														Total
	9	10	11	12	13	14	15	16	17	18	19	20	21	22	
Mill Feed (kt)	-	251	744	1,260	1,260	1,262	1,261	1,260	1,260	1,260	1,261	1,261	1245	602	14,185
Au (g/t)	-	3.08	2.92	2.73	3.51	4.10	3.62	3.05	3.11	5.16	4.70	2.94	2.76	2.57	3.48
Au (koz)	-	25	70	111	142	166	147	124	126	209	190	119	111	50	1,589
Ag (g/t)	-	0.07	0.15	0.47	0.78	0.66	0.30	0.20	0.31	0.37	0.45	0.39	0.41	0.49	0.41
Ag (koz)	-	0.5	3.6	19.0	31.7	26.6	12.0	8.3	12.5	15.1	18.2	15.8	16.4	9.5	189.1
Cu (%)	-	0.04	0.30	0.19	0.11	0.08	0.06	0.05	0.04	0.04	0.07	0.07	0.10	0.12	0.09

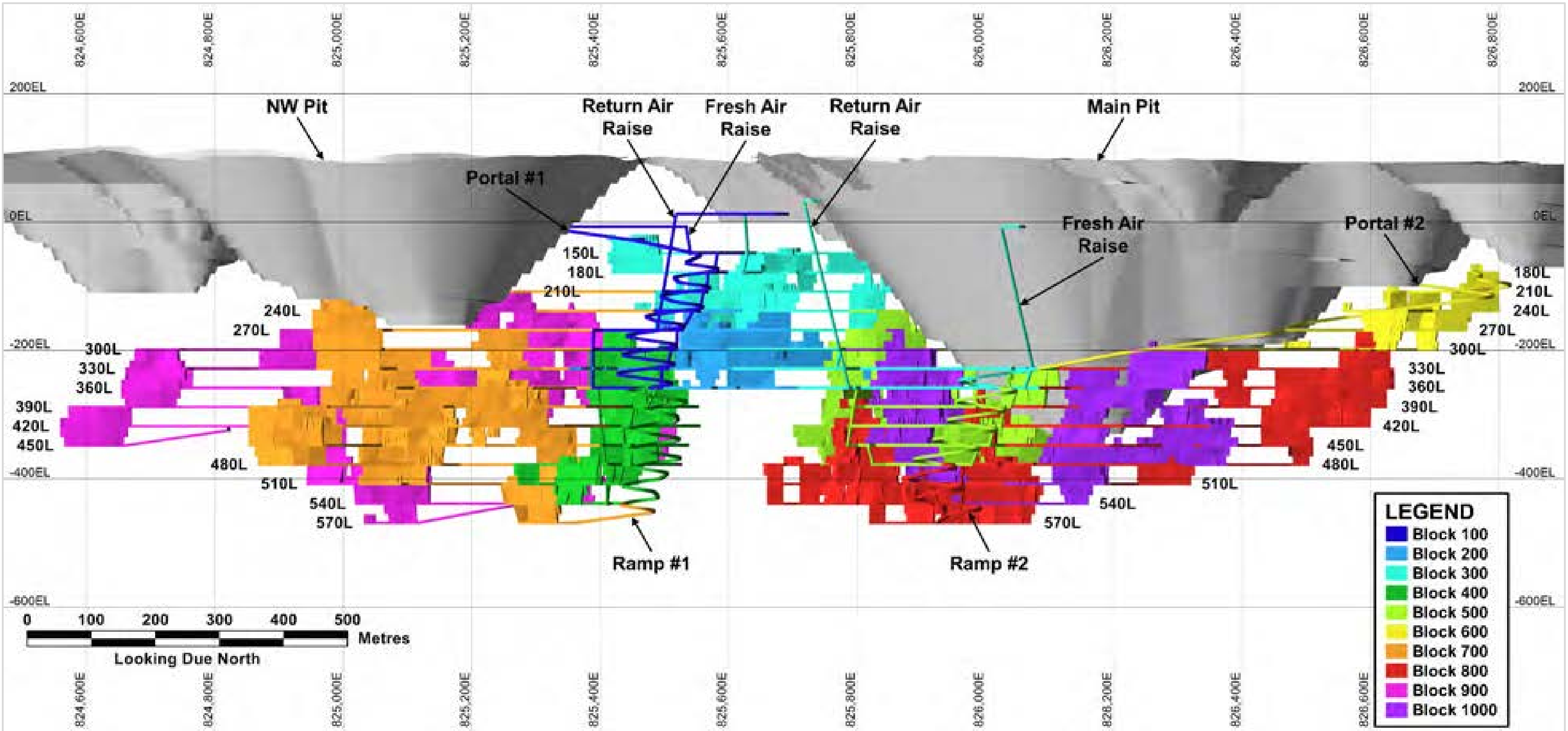


Figure 16-22: Underground mine design by block

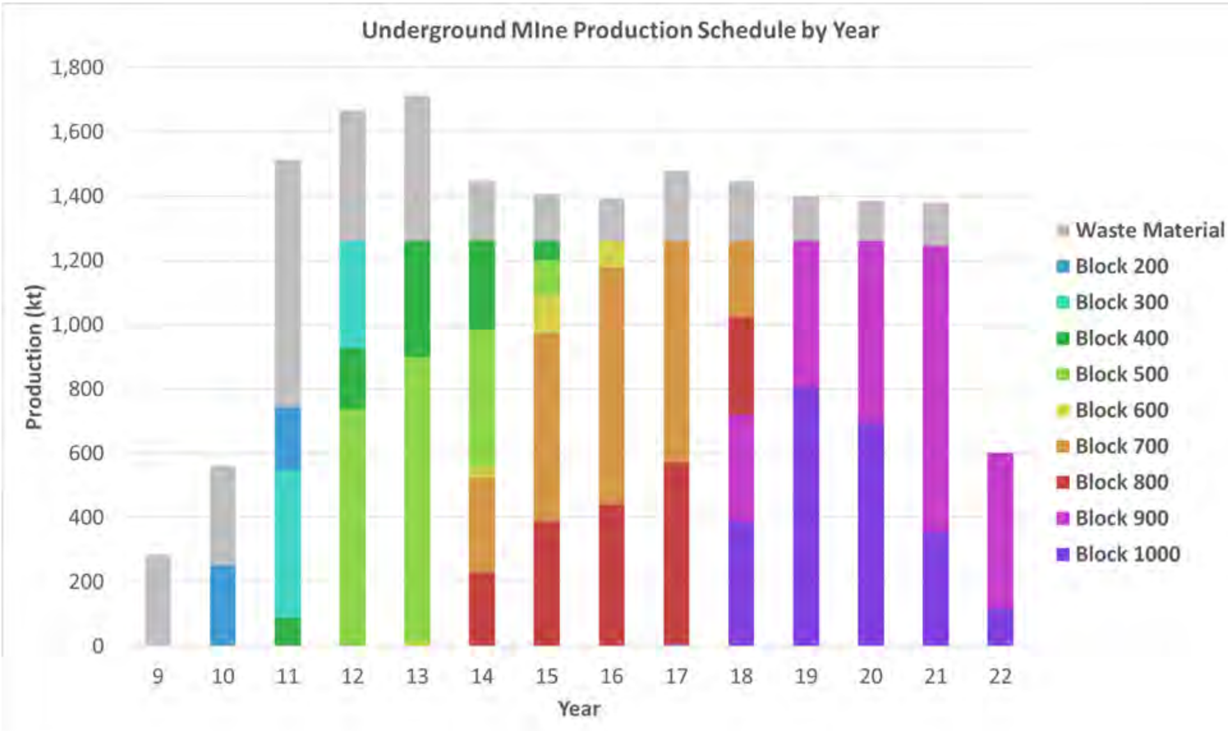


Figure 16-23: Underground mine production schedule by year

16.4.12 Ventilation

Fresh air is delivered to the underground via a combination of the two declines and two fresh air raises (FAR). Air is exhausted from the underground via two return air raises (RAR). Fresh air is delivered to production areas via auxiliary fans and flexible ducting. Table 16-40 outlines airway sizes and nominal velocities. Figure 16-24 shows the underground ventilation configuration.

Table 16-40: Airway Size and Velocities

Airway Type	Airway Size	Velocity
Fresh Air Raise	3 m ø	12 – 14 m/s
Return Air Raise	3.8 m ø	14 – 18 m/s
Decline Ramp	5 x 5 m	3 – 4 m/s

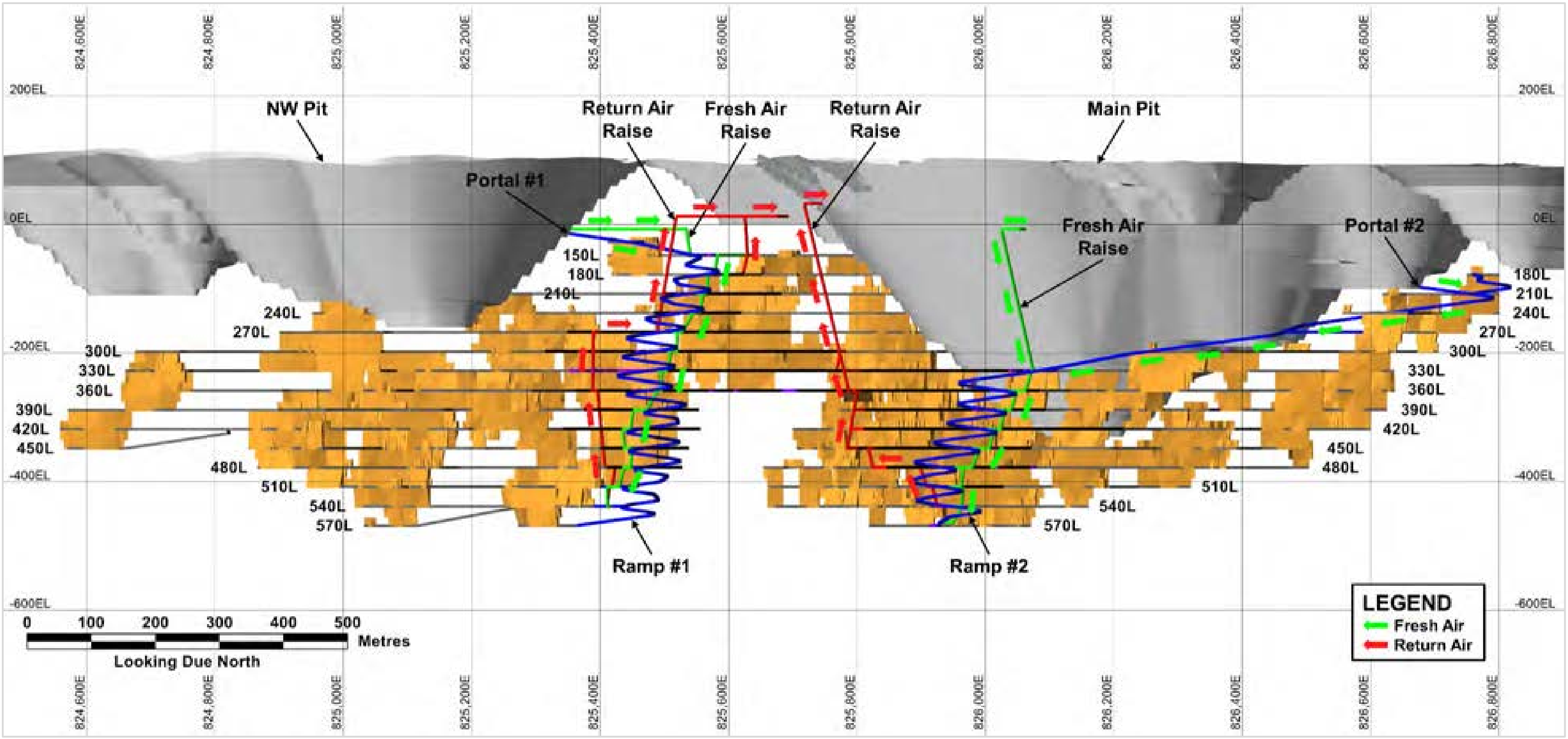


Figure 16-24: Underground ventilation configuration

The ventilation airflow requirements for the underground are based on the engine power requirements and the diesel emissions of the equipment fleet. Table 16-41 lists the diesel equipment and respective engine power and ventilation requirements. At peak productivity, the total airflow requirement is expected to be approximately 600 m^3/s (1.3 MCFM).

Table 16-41: Diesel Equipment Ventilation Requirements

Underground Mobile Fleet	Diesel Utilization (%)	Engine Power (kW)	Vent Required (m3/s)
Haul Truck (45t)	95%	447	25.5
LHD (15t)	80%	298	14.3
LHD (10t)	80%	186	8.9
Drill Jumbo	60%	180	~0
Bolter	80%	65	~0
Production Drill ITH	70%	105	~0
Production Drill TH	70%	105	~0
Scissor Lift	25%	130	1.9
ANFO Loader	25%	97	1.5
Emulsion Loader	15%	103	0.9
Shotcrete Unit	50%	103	3.1
Grouting Unit	20%	50	0.6
Fuel/Lube Truck	75%	130	5.8
Utility/Crane Truck	75%	130	5.8
Boom Truck	25%	130	1.9
Grader	95%	216	12.3
Personnel Carrier	10%	95	0.6
Pick-up Truck	25%	95	1.4

In years 10 and 11, as production commences in Block 200 and 300, fresh air will be provided by the ramp decline #1 and central FAR and exhausted via the central RAR. As production progresses to block 400 at depth, the central FAR and RAR are extended to accommodate ventilation requirements. In year 12, the eastern FAR and RAR are brought online to accommodate the additional ventilation demands of Block 500 production, and the eastern portal #2 and ramp decline #2 are commenced. In year 14, the eastern decline connects to the central portion of the mine, allowing for additional ventilation for Block 600 to begin production. The eastern FAR and RAR are also extended at depth to accommodate Block 800 production. Year 14 marks peak productivity and maximum airflow requirement of approximately 600 m^3/s (1.3 MCFM). Table 16-38 shows the schedule by block and year.

16.4.13 Dewatering

The mine dewatering system is designed to accommodate the groundwater inflows from the portals and mine workings, along with inflows from drill and other underground operating equipment. The dewatering system includes small sumps on each production level and two sets of larger main sumps. Each larger sump is designed to accommodate a 1,500 gallons per minute (gpm) capacity, for a total of 3,000 gpm mine wide. Figure 16-25 shows a long section of the mine infrastructure, including the locations of sumps.

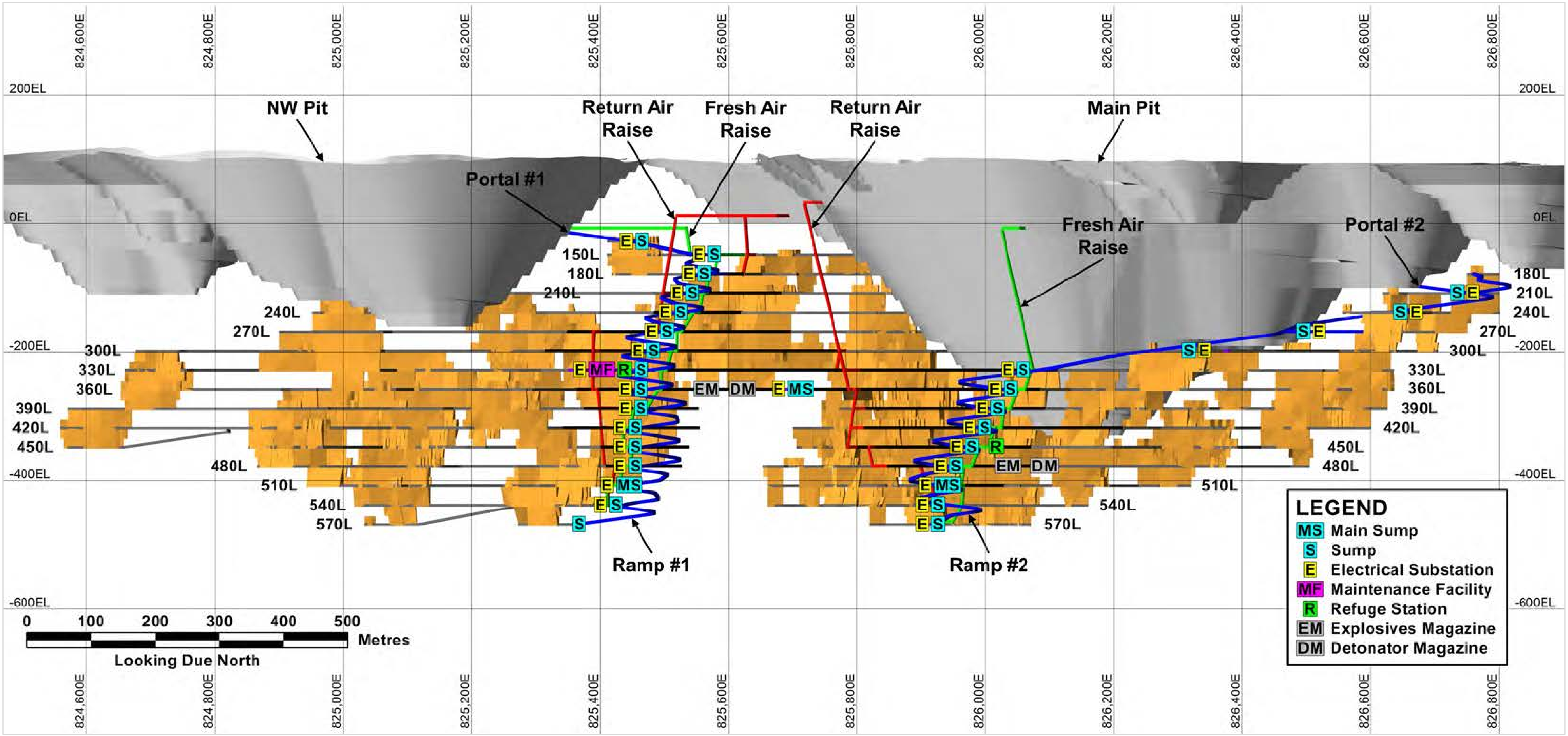


Figure 16-25: Underground infrastructure

16.4.14 Maintenance Facility

The underground includes one main underground maintenance facility on 330L. The maintenance facility is expected to provide assistance for routine maintenance (e.g., PMs) and unexpected maintenance (e.g., flat tires) for the underground fleet. Large scope maintenance needs (e.g., rebuilds) are expected to be conducted via the surface maintenance facilities. Figure 16-25 shows a long section of the mine infrastructure, including the location of the underground maintenance facility.

16.4.15 Explosives Storage

The underground mine explosives are stored in underground magazines, providing the capacity required for development and production requirements. Explosives are delivered underground on an as needed basis via an on site surface explosives storage area, which is supplied by vendor deliveries to site. The Underground powder and primer magazines are located on 360L and 480L. The 360L magazines are designed for use for Blocks 100, 200, 300, 400, 700 and 900 and the 480L magazines are designed for use for Blocks 500, 600, 800 and 1000. Figure 16-25 shows a long section of the mine infrastructure, including the locations of the underground powder and primer magazines.

16.4.16 Health and Safety

Mine Safety and Health Administration (MSHA) safety standards are incorporated in the mine design and include dual secondary means of mechanical egress. The mine communication system will have both mine phones and wireless communication through a leaky feeder system. A mine rescue team will be required to support the mine’s underground operations. The mine safety program will integrate with local providers in case of any mine emergency. Additionally, a stench gas emergency warning system will be installed in all intake ventilation systems. This system can be activated to warn underground employees of a fire situation or other emergency whereupon emergency procedures will be followed.

Two permanent refuge stations are located on the 330L haulage (near to the maintenance shop) and on 450L haulage. In additional to the two permanent refuge stations, due to the widespread nature of the underground, portable refuge stations have been included in the underground design. Portable refuge stations can be transported close to working areas and are housed in existing disused cut-outs. Each refuge chamber will be sufficiently equipped to house 12 or more persons, depending on location and unit size for up to 36 hours. The stations are self-sufficient and include seating, a chemical toilet, emergency food and water, back up power, lighting, and communications via external antenna and power supply. The breathable air system that is incorporated within the refuge chambers includes a standard compressed air line tie-in, oxygen cylinders connection, as well as an oxygen candle. Each chamber can be located at the most strategic location as dictated by the mining operation and underground workings. The chambers are easily transported by forklifts or LHD units. Figure 16-25 shows a long section of the mine infrastructure, including the locations of permanent refuge stations.

Underground dust suppression is achieved primarily by reducing airborne dust particulate with the use of wetting down muck piles, water sprays in blast headings and water atomizers in the main ramps.

16.4.17 Mine Service Distribution System

The underground mine will be supplied with two air compressors and be equipped with a leaky feeder system that will allow phone and radio communications underground, as well as standard underground call phones with intercom.

Permanent electrical substations are located on each mining level, place near to the ramp access. The permanent electrical substations are designed to provide power to level infrastructure (e.g., sumps, maintenance facility, refuge stations, ventilation fans). Temporary electrical substations are included in design as a supplement to the permanent electrical substations. Temporary electrical substations are designed to provide power to active development and production areas, which are located at distance from the ramp access. The temporary electrical substations are placed in disused crosscuts and can be relocated, as necessitated by the development and production schedule. It is expected that the underground will require a total 28 permanent and ten temporary substations.

16.4.18 Workforce

The expected workforce for the underground mine will average 320 people, including technical service staff and management. The workforce is estimated based on the production schedule and equipment requirements and includes a combination of local skilled labour with experienced technical service staff and management.

Underground workforce is scheduled on three separate rosters; 8 days on site / 6 days off site, 14 days on site / 7 days off site, and 6 weeks on site / 3 weeks off site. The underground operations and maintenance hourly workforce is scheduled on a 14 days off site / 7 days off site rotation. The underground technical service and maintenance staff are scheduled on a combination of the three rotations, with senior staff commonly working the 6 weeks on site / 3 weeks off site and junior staff commonly working 8 days on site / 6 days on site. Surveyors and technicians are expected to work closely with the underground hourly workforce and are scheduled on a 14 days on site / 7 days off site.

Table 16-42, Table 16-43, Table 16-44, Table 16-45, and Table 16-46 outline the underground management and technical staff, underground maintenance staff, underground operations hourly workforce, and underground maintenance hourly workforce requirements, respectively.

Table 16-42: Yearly Underground Workforce Count by Type

Underground Workforce Type	Schedule Year														
	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22
Management & Technical Staff	10	29	49	58	58	58	53	53	53	51	51	51	49	44	36
Maintenance Staff	-	2	7	9	9	9	9	9	9	9	9	9	9	9	9
Operations Hourly	-	63	124	205	229	232	202	202	193	193	193	190	190	160	109
Maintenance Hourly	-	17	36	60	60	60	60	60	60	60	60	60	60	60	60
Total Workforce	**10**	**110**	**216**	**332**	**356**	**359**	**324**	**324**	**315**	**313**	**313**	**310**	**308**	**273**	**214**

Table 16-43: Yearly Underground Management and Technical Staff Workforce Count

Management & Technical Staff	Schedule Year														
	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22
Underground Mine Manager	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Underground Mine General Foreman	1	2	3	5	5	5	4	4	4	4	4	4	4	4	3
Chief Mining Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Senior Mining Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Long Term Planning Engineer	-	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Short-Term Planning Engineer	-	-	1	2	2	2	2	2	2	2	2	2	2	2	2
Stope Designer Engineer	-	-	2	3	3	3	3	3	3	3	3	3	3	3	2
Backfill Engineer	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1
Ventilation Engineer	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Ventilation Technician	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1
Surveyors	-	3	6	6	6	6	4	4	4	4	4	4	4	4	4
Technician	-	1	1	2	2	2	2	2	2	2	2	2	2	2	2
Senior Geotechnical Engineer	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Geotechnical Engineer	-	-	1	2	2	2	2	2	2	2	2	2	2	2	2
Geotechnical Technician	-	-	-	2	2	2	2	2	2	2	2	2	2	2	1
Chief Mine Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Senior Mine Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Beat Geologist	-	2	4	5	5	5	5	5	5	5	5	5	5	3	3
Senior Modelling Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Infill Drilling Supervisor	-	1	2	2	2	2	2	2	2	2	2	2	1	1	-
Senior Field Logging Geologist	-	1	2	2	2	2	2	2	2	1	1	1	1	1	-
Core Logger	-	2	6	6	6	6	4	4	4	3	3	3	2	2	-
Project Lead	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mechanical Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Civil Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Underground Safety Coordinator	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Clerk	-	3	5	5	5	5	5	5	5	5	5	5	5	5	4
Total Management & Technical Staff	**10**	**29**	**49**	**58**	**58**	**58**	**53**	**53**	**53**	**51**	**51**	**51**	**49**	**44**	**36**

Table 16-44: Yearly Underground Maintenance Staff Workforce Count

Maintenance Staff	Schedule Year														
	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22
Maintenance Superintendent	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance General Foreman	-	1	1	2	2	2	2	2	2	2	2	2	2	2	2
Maintenance Planning Coordinator	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2
Maintenance Planning Engineer	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2
Maintenance Planning Technician	-	-	1	2	2	2	2	2	2	2	2	2	2	2	2
Total Maintenance Staff	**-**	**2**	**7**	**9**	**9**	**9**	**9**	**9**	**9**	**9**	**9**	**9**	**9**	**9**	**9**

Table 16-45: Yearly Underground Operations Hourly Workforce Count

Operations Hourly	Schedule Year														
	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22
Development Supervisor	-	3	3	3	3	3	3	3	3	3	3	3	3	3	-
Jumbo Operator	-	6	6	12	9	9	6	6	6	6	6	6	6	3	-
Bolter Operator	-	5	9	15	15	15	12	12	12	12	12	12	12	6	-
Cablebolting / Grouting	-	3	6	6	6	6	3	3	3	3	3	3	3	3	-
Service Crew	-	5	9	12	12	12	9	9	6	6	6	6	6	3	3
Service Crew Helper	-	5	9	12	12	12	9	9	6	6	6	6	6	3	3
Production Supervisor	-	-	2	6	6	6	6	6	6	6	6	6	6	6	3
Truck Driver	-	6	9	18	24	27	24	24	24	24	24	21	21	21	12
LHD Operator	-	6	18	21	24	24	21	21	21	21	21	21	21	21	15
Production Drill Operator	-	-	3	6	9	9	9	9	9	9	9	9	9	9	6
Lead Blaster	-	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Blaster Helper	-	3	6	9	9	9	6	6	6	6	6	6	6	6	6
Grader Operator	-	3	3	3	6	6	6	6	6	6	6	6	6	6	6
Fuel / Lube Truck Operator	-	3	3	6	6	6	6	6	6	6	6	6	6	6	6
Utility / Labourer / Nipper / Helper	-	5	9	21	30	30	24	24	21	21	21	21	21	18	12
Underground Pastefill & Construction Supervisor	-	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Pastefill Piping Crew	-	-	6	12	12	12	12	12	12	12	12	12	12	6	3
Pastefill Barricade Crew	-	-	-	3	6	6	6	6	6	6	6	6	6	3	3
Pastefill Pour Watcher	-	-	-	3	3	3	3	3	3	3	3	3	3	3	3
Shotcrete Operator	-	3	3	6	6	6	6	6	6	6	6	6	6	6	6
Construction Crew	-	3	6	9	9	9	9	9	9	9	9	9	9	6	3
Backfill Plant Supervisor	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1
Backfill Plant Operator	-	-	2	3	3	3	3	3	3	3	3	3	3	3	3
Backfill Plant Helper	-	-	2	3	3	3	3	3	3	3	3	3	3	3	3
Binder Transport & Delivery Operator	-	-	5	9	9	9	9	9	9	9	9	9	9	9	6
Total Operations Hourly	**-**	**63**	**124**	**205**	**229**	**232**	**202**	**202**	**193**	**193**	**193**	**190**	**190**	**160**	**109**

Table 16-46: Yearly Underground Maintenance Hourly Workforce Count

Maintenance Hourly	Schedule Year														
	8	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**	**20**	**21**	**22**
Underground Shop Supervisor	-	-	-	3	3	3	3	3	3	3	3	3	3	3	3
Surface Truck Shop Supervisor	-	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Underground Shop Mechanic	-	-	-	6	6	6	6	6	6	6	6	6	6	6	6
Underground Millwright	-	-	-	3	3	3	3	3	3	3	3	3	3	3	3
Underground Shop Electrician	-	-	-	6	6	6	6	6	6	6	6	6	6	6	6
Underground Shop Welder	-	-	-	6	6	6	6	6	6	6	6	6	6	6	6
Underground Shop Mechanic Helper	-	-	-	6	6	6	6	6	6	6	6	6	6	6	6
Underground Shop Electrician Helper	-	-	-	3	3	3	3	3	3	3	3	3	3	3	3
Underground Shop Welder Helper	-	-	-	3	3	3	3	3	3	3	3	3	3	3	3
Surface Millwright	-	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Surface Shop Mechanic	-	5	9	3	3	3	3	3	3	3	3	3	3	3	3
Surface Shop Electrician	-	3	6	3	3	3	3	3	3	3	3	3	3	3	3
Surface Shop Welder	-	3	6	3	3	3	3	3	3	3	3	3	3	3	3
Surface Shop Mechanic Helper	-	-	3	3	3	3	3	3	3	3	3	3	3	3	3
Surface Shop Electrician Helper	-	-	3	3	3	3	3	3	3	3	3	3	3	3	3
Surface Shop Welder Helper	-	-	3	3	3	3	3	3	3	3	3	3	3	3	3
Total Maintenance Hourly	**-**	**17**	**36**	**60**	**60**	**60**	**60**	**60**	**60**	**60**	**60**	**60**	**60**	**60**	**60**

16.4.19 Mobile Equipment

Equipment requirements were calculated from production rates and typical availabilities for equipment in underground mines. The equipment replacement schedule for haul trucks, LHDs, drill jumbos, bolters, production drills, graders, personnel carriers, and pick-up trucks is set at 6 years. The equipment replacement schedule for all other underground mobile equipment is set at 9 years. Table 16-47 outlines the number of underground mobile equipment required by year.

Table 16-47: Underground Mobile Equipment Required by Year

Underground Mobile Equipment	Schedule Year													
	9	10	11	12	13	14	15	16	17	18	19	20	21	22
Haul Truck	2	3	7	9	10	9	9	9	9	9	8	8	8	4
LHD 10 yd	-	3	3	4	4	4	4	4	4	4	4	4	4	4
LHD 6 yd	2	3	5	5	5	4	4	4	4	4	4	4	4	2
Drill Jumbo	2	2	4	3	3	2	2	2	2	2	2	2	1	-
Bolter	2	3	5	5	5	4	4	4	4	4	4	4	2	-
Production Drill ITH	-	1	1	2	2	2	2	2	2	2	2	2	2	1
Production Drill Top-hammer	-	1	2	2	2	2	2	2	2	2	2	2	2	2
Scissor Lift	2	3	4	4	4	4	4	4	4	4	4	4	3	3
ANFO Loader	2	2	3	3	3	3	3	3	3	3	3	3	2	2
Emulsion Loader	-	1	1	1	1	1	1	1	1	1	1	1	1	1
Shotcrete Unit	1	1	2	2	2	2	2	2	2	2	2	2	2	2
Grouting Unit	1	1	2	2	2	2	2	2	2	2	2	2	2	2
Fuel / Lube Truck	1	1	2	2	2	2	2	2	2	2	2	2	2	2
Utility / Crane Truck	1	2	3	3	3	3	3	3	3	3	3	3	3	2
Boom Truck	-	1	1	1	1	1	1	1	1	1	1	1	1	1
Grader	1	1	1	2	2	2	2	2	2	2	2	2	2	2
Personnel Carrier	2	5	10	10	10	10	10	10	10	10	10	10	10	8
Pick-up Truck	6	8	12	12	12	12	12	12	12	12	12	12	12	10

17 RECOVERY METHODS

17.1 Summary

The various ore types to be processed include:

- Saprolite materials (SAP).
- Low copper-gold or gold only hardrock (LCO).
- Hardrock high copper and/or copper-gold ores to be processed by flotation (ACO).
- Material that is suited to processing via flotation or direct gold processing methods given the net revenue is similar (FLEX).

The Gold Plant (built as the initial plant) process flowsheet includes:

- Ore receipt crushing and coarse ore stockpile.
- Reclaim and SAG milling including a gravity gold circuit.
- Intensive cyanidation of gravity concentrate.
- Leaching of the fine slurry using the (hybrid) CIL process.
- Recovery of the loaded carbon and elution to recover gold and silver values followed by carbon reactivation.
- Electrowinning gold and silver and smelting the doré product.
- Tailings thickening and cyanide detoxification process prior to the slurry being transferred to tailings storage.

The Flotation Plant (built as an expansion and operational in Year 6) process flowsheet includes:

- Ore receipt, 3 stage crushing and fine ore stockpile.
- Reclaim and ball milling including a gravity gold circuit.
- Intensive cyanidation of gravity concentrate.
- Flotation circuit for the recovery of gold-copper (and silver) concentrate.
- Concentrate dewatering and filtration of final gold-copper concentrate.
- Tailings thickening prior to the slurry being transferred to tailings storage.

The concentrator is designed to process 14,000 tpd of mineralized material (nominal) in total at its peak operation. During the first 5 years of the Project, the plant will be fed a consistent blend of LCO and SAP material to the Gold Plant. The total feed to the plant during this time will be 7,000 tpd.

In year 6, ACO material will be processed through a parallel 7,000 tpd Cu Flotation Plant producing an Au and Ag bearing Cu concentrate, thereby doubling the on site plant capacity to 14,000 tpd.

Fundamentally, the gold and flotation circuits are stand alone apart from sharing select services and some flotation by-products are directed to the CIL for leaching of contained precious metal values.

17.2 Flowsheet Selection

17.2.1 Gold Plant

The material to be processed comprises a blend of saprolite, minor transitional material and hardrock with saprolite and hardrock being dominant.

The production schedule commences in the first three years of operation with a blend 30% of saprolite in the total feed, the remainder of which is predominantly fresh hardrock. In the fourth year the blend is around 20% saprolite to hardrock and from year five onwards, the blend is 15% saprolite to hardrock. Saprolite is exhausted as the end of mine life approaches.

The original flowsheets for the project assumed a separate saprolite processing front-end to manage the sticky/adhesive and potentially viscous properties of saprolite as a dominant and periodic feed source. As a consequence of the revised processing schedule, materials handling issues are anticipated to be reduced given the ability to continuously “scour” bins, transfer chutes and hoppers of sticky oxides with the addition of rock. This in turn means that a dedicated saprolite circuit is not necessary.

While the saprolite can be managed by some unit processes, the ratio of saprolite was still considered problematical for a three-stage crushing circuit. With the recent successes of large single stage SAG milling circuits (SSAG), it was considered appropriate to implement this type of circuit which has the ability to handle saprolite based feeds as well as hardrock.

Consideration of the high rainfall location and the impact this can have on plant operations must also be given. A SSAG circuit offers advantages under these conditions.

Consequently, the flowsheet selection has assumed that saprolite will not dominate the blends. Therefore, the ore receipt, crushing and milling processes, equipment and style of plant were selected with this in mind.

The downstream circuit is considered conventional for a free milling ore. A gravity circuit is proposed to recovery coarser gold from the recirculating load in the grinding circuit. The grinding circuit product is then leached, and gold and silver values adsorbed onto activated carbon. The leaded material is thickened, the cyanide content detoxified, and the resulting slurry pumped to the TSF.

The leach-adsorption process selected is a hybrid CIL configuration. This process presents a lower capital cost entry point compared to other configurations as well as utilizing a leach tank to elevate solution grades and subsequent activated carbon loadings.

The activated carbon is recovery and eluted of the gold and silver values using an AARL elution circuit and the metal values electroplated prior to smelting to produce doré for sale.

The selection of the cyanide detoxification process was based on the need to provide both environmental protection as well as reduce impact on the processing plant performance due to the recirculation of cyanide species. The SO_2/air process being selected, being the most common, well proven, and effective process for this type of facility.

The AARL process was selected for elution duties as this process provides for elevated elution rates if necessary to deal with high cyanide soluble copper loads that are expected from time to time. The process is also easy to automate to include a cold cyanide wash to reduce the impacts of cyanide soluble copper in the elution process proper.

17.2.2 Flotation Plant

The flotation plant will only process hardrock ore types and as such, the issues of saprolite handling are negated.

The original flotation flowsheets proposed for the project included three-stage crushing feeding a ball mill to deal with the abrasion characteristics of the ore feeds. This circuit has been retained for the purposes of the PEA and the capital and operating costs are based on such a configuration.

Such a circuit provides a consistent feed gradation to the ball mill which in turn can be expected to provide a consistent grind product to flotation.

A gravity circuit is included in the ball mill recirculating load to capture coarse free gold values. This is of additional value in that this gold will report to doré and will command a higher payment than if these gold credits were to report to the copper concentrate.

The grinding circuit product reports to a conventional roughing-regrind-cleaning type flowsheet for flotation. Roughing (post conditioning of the ore) is conducted at a primary grind of nominally P_{80} 150 µm. The rougher concentrate is then reground in a stirred mill prior to cleaning. This configuration allows for savings in grinding power in that it is only the rougher concentrate that has to be fine ground and not the complete flotation feed.

Minor gold values are contained in the rougher tails. As a result, a rougher scavenger circuit is included, the concentrate from which is directed to the CIL for leaching via a thickener. The rougher scavenger tail is the final tail from the circuit. It is thickened and pumped to the TSF.

The cleaning circuit includes two stages of cleaning.

The tail from the first set of cleaners is directed to the flotation products thickener with the rougher scavenger concentrate for leaching in the CIL

The first stage of cleaning concentrate is directed to a second stage of cleaning where the concentrate is classed as final concentrate. The second stage cleaner tail is sent for regrinding and recirculated to the first stage of cleaning.

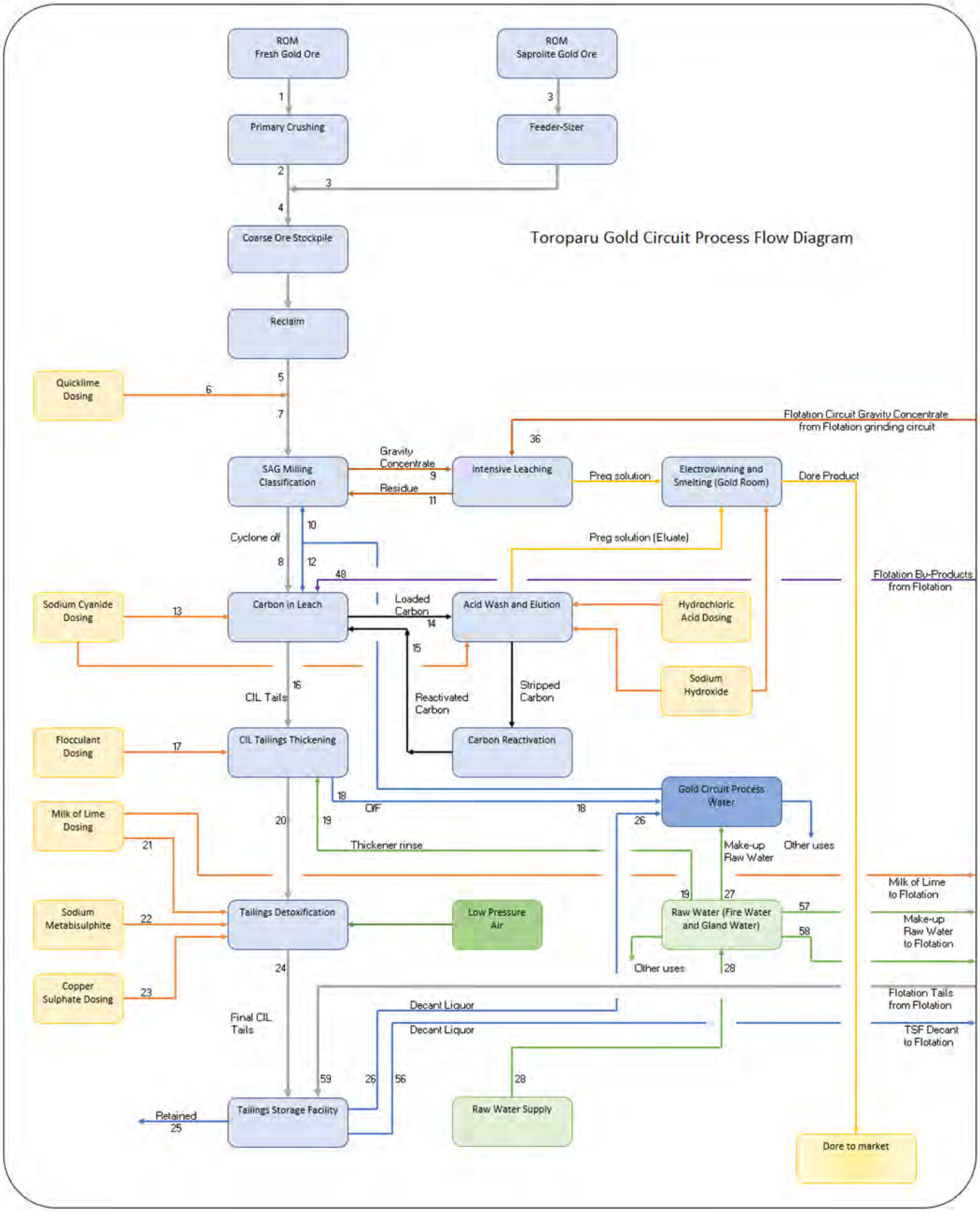
Following cleaning, the concentrate is thickened and then pressure filtered to provide a concentrate to be sent off site in containers to market. A pressure filter selected to ensure the transportable moisture limit (TML) is met given solar drying options on this high rainfall site are limited.

17.3 Processing Description

The process flowsheet for this study is presented as Figure 17-1 for the gold plant and Source: Metifex, 2021

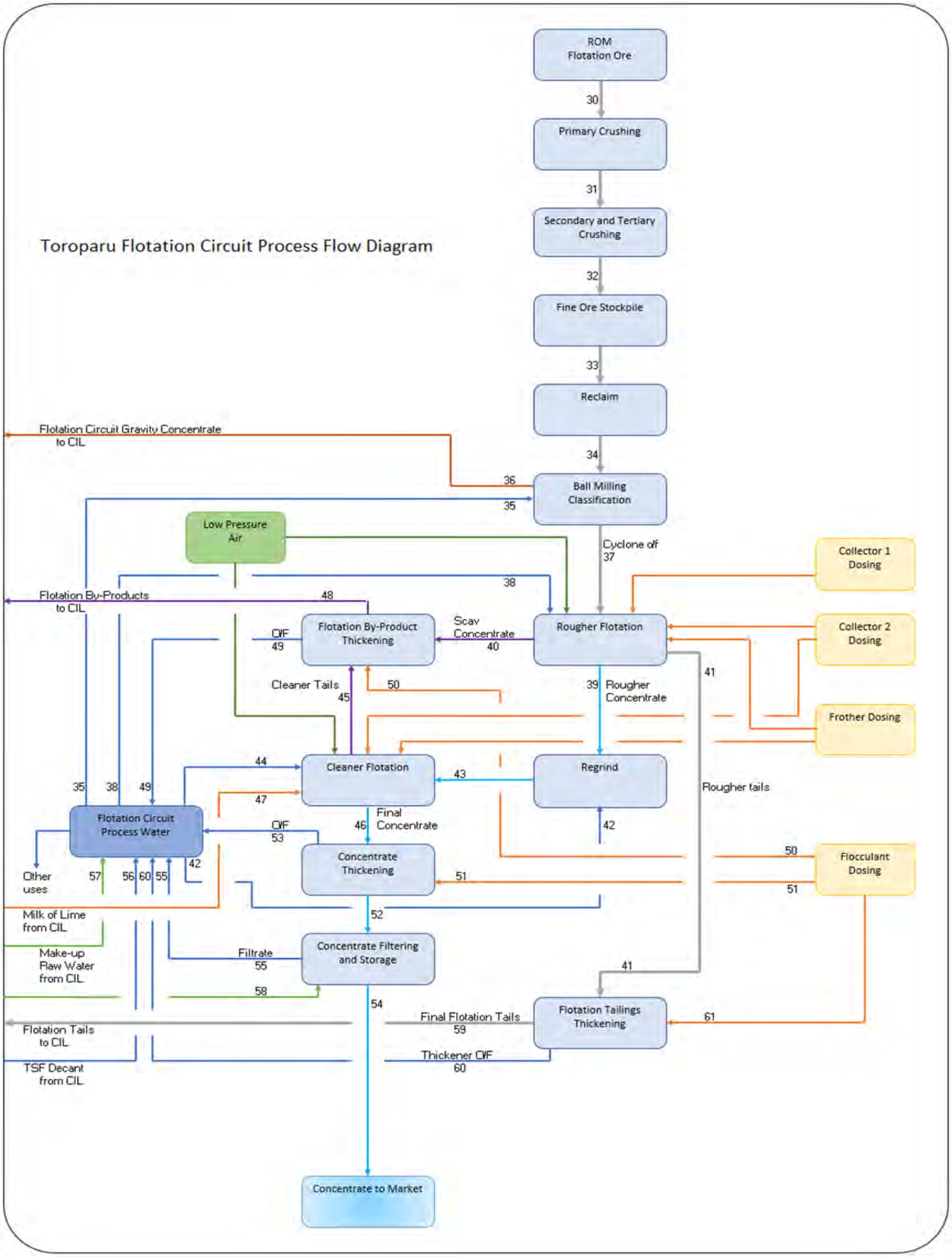
Figure 17-2 for the flotation plant. Interfacing connectors between the two flowsheets are shown. These will only be relevant when the two facilities are operational.

The stream descriptions for the numbered streams are presented per Table 17-1.



Source: Metifex, 2021

Figure 17-1: Process flow diagram – gold ore processing



Source: Metifex, 2021

Figure 17-2: Process flow diagram – flotation ore processing

Table 17-1: Flowsheet Stream Descriptions

Stream	Gold Circuit
1	Fresh ore ROM feed - Gold Circuit
2	Primary Jaw discharge
3	Saprolite Sizer discharge
4	Crushed product to Coarse Ore Stockpile
5	Reclaim feed to SAG mill
6	Quicklime dosing to SAG feed
7	SAG mill feed
8	Cyclone overflow to CIL
9	Gravity concentrate to intensive leach
10	PW to Grinding, Gravity and Classification
11	Gravity concentrate from intensive leach to SAG mill
12	PW dilution to CIL
13	Sodium Cyanide to CIL
14	Loaded Carbon to Elution
15	Reactivated (barren) Carbon to CIL
16	CIL tails to Thickening
17	Flocculant Dosing to CIL Tails Thickening
18	CIL Tails Thickener overflow
19	RW to CIL Tails Thickener (rinse)
20	CIL Tails Thickener underflow
21	Milk of Lime dosing to Tails Detoxification
22	Sodium Metabisulphite dosing to Tails Detoxification
23	Copper Sulphate dosing to Tails Detoxification
24	Tailings Detoxification product to TSF
25	Solids/Solution retained in TSF
26	Decant to Gold Circuit PW Pond.
27	Make-up RW to Gold Circuit PW Pond
28	RW supply to site

Stream	Flotation Circuit
30	Fresh ore ROM feed - Flotation Circuit
31	Primary Jaw discharge
32	Secondary and Tertiary Crushing product
34	Reclaim feed to Ball mill
35	PW to Grinding, Gravity and Classification
36	Gravity concentrate from Flotation
37	Cyclone overflow to Flotation
38	PW to Rougher Flotation
39	Rougher Concentrate to Regrind
40	Scavenger Concentrate
41	Flotation Tailings to Thickening
42	PW to Regrind
43	Regrind product to Cleaners
44	PW to Cleaners
45	Cleaner Tails to By-product Thickener
46	Final Cleaner Concentrate to Conc Thickener
47	Milk of Lime dosing to Cleaners
48	Flotation By-Products to CIL
49	By-Product Thickener overflow
50	Flocculant to By-Products Thickener
51	Flocculant to Concentrate Thickener
52	Concentrate Thickener underflow
53	Concentrate Thickener overflow
54	Filtered Product to Load-Out
55	Filtrate
56	Decant to Float Circuit PW Pond
57	Make-up RW to Flotation Circuit PW Pond
58	Filter cake wash
59	Flotation Tailings to TSF
60	Flotation Tailings Thickener overflow
61	Flocculant to Float Tails Thickener

The process descriptions are presented separately for the Gold and Flotation Plants.

17.3.1 Gold Plant

17.3.1.1 Crushing

Ore is fed to a RoM hopper by front-end loader. Ore is recovered by an apron feeder and is scalped on a vibrating grizzly prior to the grizzly oversize being jaw crushed. The fines and crushed product are recombined and conveyed to a coarse ore stockpile.

17.3.1.2 Reclaim, Grinding and Gravity

Reclaim of coarse crushed ore is by dual apron feeders. Ore is conveyed to the SAG mill feed chute. Quicklime is dosed directly to the SAG feed conveyor to provide pH control in the mill and downstream CIL circuit.

Ore enters the SAG mill and is combined with process water, cyclone underflow and gravity circuit tailings. Upon exit of the SAG mill, the slurry is screened over a vibrating discharge screen to remove pebbles. The SAG mill will be variable speed.

The pebbles recirculate via a pebble crusher feed bin and crusher. Crushed pebbles are discharged onto the SAG feed conveyor and re-introduced to the SAG mill. A cross-belt magnet and metal detector along with a flop-gate for bypass provide pebble crusher protection.

An emergency feed bin is provided adjacent to the pebble crusher. Coarse crushed ore can be reclaimed direct from the stockpile and fed via this bin as needed to cater for reclaim feeder shut-downs. Grinding media can be added to the emergency feed bin with some coarse crushed material to allow media to be fed to the mill.

The fines from the SAG discharge screen gravitate to the cyclone feed hopper. Process water is added, and the slurry pumped to the cyclones. Cyclone overflow reports via a trash screen to the CIL circuit feed box for leaching. The trash screen removing mis-reporting mineral and also low specific gravity trash such as vegetation and blasting debris. The function of the trash screen being to ensure the carbon circuit is not contaminated with oversize particles.

The cyclone underflow gravitates to a splitter directing the slurry to either the SAG feed chute or nominally one third to a gravity scalping screen. The gravity screen undersize is directed to a centrifugal gravity concentrator, the tails from which gravitate to the SAG feed chute. Coarse gravity screen oversize is directed to the SAG feed chute.

Gravity concentrate is periodically discharged to a holding tank and then to an intensive leach reactor. Here the concentrate is leached, and the pregnant liquor is subjected to electrowinning to recover precious metals (in the gold room).

Gravity circuit equipment is not process critical and as such, no standby systems will be provided.

The intensive cyanidation tails are redirected back to the SAG mill discharge hopper.

17.3.1.3 CIL

Protective alkalinity of the slurry (to protect the cyanide and reduce losses of cyanide to the atmosphere) is achieved by dosing quicklime to the SAG feed conveyor. Monitoring of pH in the CIL is used to control the quicklime dose or supplementation by caustic soda if necessary. Sodium cyanide is dosed via the CIL feed box to dissolve the gold and silver in the ore. Trimming doses can be applied down the CIL tankage as needed via additional dosing points from the cyanide ring main.

The first CIL tank is used as a dedicated leach tank. Facility is provided to install an inter-tank (carbon retention screen) if required. However, typically the first tank will only be used for leaching as this increases the solution tenor which in turn increases activated carbon loadings, reducing elution capacity demands.

Low pressure air is sparged into the last CIL tanks to provide oxygen for leaching. Oxygen is sparged to the first four CIL tanks to offer accelerated leaching conditions and potentially reduce cyanide consumption. A flow on benefit regarding cyanide detoxification demands is often a result of this process detail.

The slurry flows to the second tank (first adsorption tank) which contains activated carbon. The carbon adsorbs the gold and silver cyanide species from the liquor and at the same time effectively purifies and concentrates these metals by a factor of around 1,000 times for gold and less so for silver. Leaching continues in the second tank and the pH and cyanide levels are monitored and maintained.

The partially leached slurry flows down through the remaining seven adsorption tanks, with gold and silver leaching continuing as the slurry progresses. These tanks also contain activated carbon which continues to adsorb the gold and silver as it is leached. At the end of the adsorption stages the slurry discharges and passes to the tailings thickener.

Each of the adsorption stages contains an inter-tank screen. The screen is nominally 1 mm in aperture. This is coarse enough to let the slurry pass but is too fine to allow the carbon to pass. The carbon is therefore retained in each of the adsorption tanks and controlled in this way.

The highest grade carbon is that in the first adsorption tank as it comes into contact with the highest grade gold and silver solutions. From the first adsorption tank, a portion of the slurry is pumped out over a loaded carbon recovery screen. This screen is also around 1 mm aperture. Loaded carbon is recovered from the slurry and captured. The slurry itself flows thought the screen and back to the first adsorption tank.

The loaded carbon is sent to elution/reactivation.

Carbon and slurry from the second adsorption tank is pumped up and into the first adsorption tank by means of a recessed impeller pump. The inter-tank screen retains this back-mixed carbon. However, the slurry can pass the screen and returns to the second adsorption tank and so on down the adsorption tank train. In this way the carbon is effectively transferred upstream into the first adsorption tank to replace that carbon that was recovered and sent to elution.

Similarly, slurry and carbon from the third adsorption tank are transferred to the second adsorption tank. Again, the slurry returns but the carbon is trapped. This process is continued from the fourth to the third tank, the fifth to the fourth and so on such that effectively the carbon flows counter-current to the flow of the slurry. The carbon is always being introduced to higher and higher-grade solutions as it is pumped forward, thereby increasing the loading.

17.3.1.4 Elution and Reactivation

The loaded carbon from the first adsorption tank is first sent to acid washing. Here dilute hydrochloric acid is used to soak the carbon and remove acid soluble contaminants that can build up and make the carbon less effective at adsorbing gold and silver.

Once rinsed of acid, the carbon is transferred to the elution column. Here a hot solution of sodium hydroxide and sodium cyanide is passed through the carbon followed by hot water at around 130 degrees Celsius. This process desorbs the gold and silver off the carbon and returns it to the solution phase. At the same time, the metal concentration is increased as the metals are now present in a small volume of solution.

Once the carbon has been eluted (or stripped as the process is often referred to) it is sent to a rotary kiln for reactivation. Here it is heated up to nominally 750 degrees Celsius to reactivate the carbon. This barren reactivated carbon is then returned to the CIL plant and re-used. It is sent to the last adsorption tank where the solution grades of gold and silver are the lowest. As this carbon is active and has little gold and silver loaded on it, it is effective in scavenging the last of the gold and silver from solution. It is then pumped forward via the carbon transfer pumps and eventually ends up as fully loaded carbon to again be eluted and recycled.

17.3.1.5 Electrowinning and Smelting

The solution containing the precious metals from the hot elution stage is passed through an electrowinning cell where the metals are plated out as a sludge on the cell cathodes.

Similarly, the intensive cyanidation electrowinning cell produces a gold and silver sludge on the cell cathodes.

This sludge is periodically washed from the cathodes, filtered/dried and then smelted in a furnace to produce the final mine product, doré.

17.3.1.6 Tailings Thickening and Cyanide Detoxification

Once the slurry has been processed in the CIL circuit it is directed over a carbon safety screen. This screen recovers mis-reporting carbon that may have escaped the CIL via poorly seated inter-tank screens or holes in the inter-tank screen media itself.

The oversize from this screen is collected and either the coarse carbon is returned to the CIL or if the oversize is low value, discarded.

The carbon safety screen undersize is directed to the tailing thickener. Here the slurry is settled with the assistance of flocculant and excess process water recovered. This water contains residual cyanide and minor precious metal values and is redirected back to the grinding circuit where these values give credit.

New raw water make up to the process water circuit is also added to the thickener. This provides some washing of the cyanide and precious metal values as it effectively dilutes the overflow stream but also the thickener underflow solution, thereby minimizing losses but also reducing the operating cost of the downstream cyanide detoxification process.

Thickener underflow is directed to the cyanide detoxification tank feed box. As the tailings thickener produces a slurry having a pulp density in excess of the pulp density required for effective cyanide detoxification, it must be re-adjusted. Over thickening is a preferred action as this maximizes the return of cyanide and precious metals to the grinding circuit and minimizes the cyanide detoxification operating cost.

Sodium metabisulfite and copper sulphate are added along with dilution water to the detoxification feed. Tailings decant water is typically used for the dilution step. Air to provide oxygen is pumped into the tank via spargers located in the base and under the tank agitator. This mix oxidizes the cyanide and also generates acid as a by-product. A hydrated lime slurry is dosed to the cyanide detoxification reactor to maintain the required alkaline pH.

The slurry overflows the cyanide detoxification reactor and gravitates to the tailings pump hopper where the tailings are combined with flotation tailings when the Flotation Plant is operating.

17.3.1.7 Reagents and Services

The following reagents are required for the gold plant. Additional detail is provided per Section 17.3.3.

- Quicklime – dosed as a solid powder from a silo via rotary valve.
- Sodium Cyanide – solids dissolved in water and dosed via pump and ring main.
- Sodium Hydroxide – solids dissolved in water and dosed via pump.
- Hydrochloric Acid – dosed at the as received concentration via pump.
- Flocculant – solids dissolved in water and dosed via pump.
- Copper Sulphate – solids dissolved in water and dosed via pump.
- Sodium Metabisulphite – solids dissolved in water and dosed via pump.
- Hydrated lime – solids slurried and dosed via ring main.

The following services are provided:

- High pressure service air.
- Low pressure air for CIL aeration.
- Low pressure air for cyanide detoxification aeration.
- Oxygen generated by pressure swing adsorption (PSA).

- Raw water storage and reticulation.
- Fire water reticulation.
- Process water storage and reticulation.
- Potable water reticulation.
- Gland water reticulation.

Raw water supply to the plant, decant water return from the TSF and tailings delivery to the TSF are outside of the process plant area and are outside the scope of the process plant capital and operating cost estimates.

17.3.2 Flotation Plant

The Process Description for the Flotation Plant is based on the 2014 DFS with modifications as required for the current mine production plan and plant capacity of 7,000 tpd.

17.3.2.1 Crushing and Stockpile

There will be one three-stage crushing and screening line for the hardrock flotation material, with a nominal capacity of 7,000 tpd. The circuit is independent of the Gold Plant crushing circuit and will feed a dedicated ball mill and subsequent processing circuit. Crushing circuit availability (on stream time) is anticipated to be 6,500 operating hours per annum. No standby equipment will be installed.

Primary crushing will be carried out by a jaw crusher. A single standard head secondary cone crusher together with a tertiary short head cone crusher will both be operated in closed circuit with a twin deck vibrating screen.

RoM material will be tipped into a receiving bin dedicated to the jaw crusher. The RoM material will be extracted from the bin by an apron feeder, which will in turn feed a vibrating grizzly. Grizzly oversize (nominally +100 mm) will be crushed in the jaw crusher while undersize will bypass the crusher. The crushed product will be combined with the grizzly undersize and conveyed to the crusher double deck screen which accepts combined product from the primary, secondary, and tertiary crushers.

The screen top deck will cut at 50 mm and the bottom deck at 16 mm. Two recycle products will be produced, which will report to the secondary and tertiary crusher depending on product size. The oversize product (+50 mm) will be conveyed to the secondary crusher while the mid-size product (16 to 50 mm) will be conveyed to the tertiary crusher. Screen undersize (-16 mm with a nominal P_{80} = 8 mm) is the final crushed product and will be conveyed to the Flotation Plant stockpile.

17.3.2.2 Reclaim, Grinding and Gravity

The Flotation Plant will utilize a ball mill grinding circuit for the feed to the flotation circuit. Milling circuit availability and all downstream plant (on stream time) is anticipated to be 8,000 hours per annum. All mainstream production critical pumps as well as certain mill utility pumps will have an installed hot standby.

The milling circuit will include two apron feeders to reclaim feed from the stockpile and discharge to the conveyor that will transport the material to a fixed speed overflow ball mill. Milled product will discharge the ball mill via a trommel and then be pumped to a cluster of cyclones.

The cyclone underflow gravitates to a splitter directing the slurry to either the ball mill feed chute or a to a gravity scalping screen. The gravity screen undersize is directed to a centrifugal gravity concentrator, the tails from which gravitate to the ball mill feed chute. Coarse gravity screen oversize is directed to the ball mill feed chute.

Ball mill cyclone overflow will gravitate to the rougher flotation conditioning tank.

The gravity concentrates will be collected in a dedicated holding tank and subsequently treated in a dedicated intensive leach reactor. The pregnant solution will then be directed to electrowinning located in the Gold Room that services the Gold Plant.

The separate intensive leaching for the flotation circuit will allow for gold accounting to be attributed to the flotation feed grade.

Gravity circuit equipment is not process critical and as such, no standby systems will be provided.

Intensive leach tails will report to the SAG mill circuit and enter the Gold Plant mass balance.

17.3.2.3 Rougher Flotation Circuit

Cyclone overflow will gravitate through a metal accounting sampler to the rougher flotation conditioning tank where the collector PAX and the frothing agent MIBC will be dosed. The rougher flotation circuit will consist of rougher and rougher scavenger cells. All mainstream production critical pumps will have an installed hot standby.

The conditioner tank will overflow to the rougher flotation cells. The rougher concentrate collected from these flotation cells will be pumped through a sampler to the regrind mill feed tank. The tailings from the final rougher flotation cell will gravitate to the rougher scavenger flotation cells.

The rougher scavenger concentrate will be pumped through a sampler to the flotation by-product pre-leach thickener. Additional dosing points for PAX and MIBC will be provided for at the first rougher scavenger cell centre well.

The rougher scavenger, once thickened, will be directed to the CIL for leaching of contained gold values.

The rougher scavenger tailings will be pumped through a sampler to the flotation tailings thickener. This is the final flotation tailings.

17.3.2.4 Flotation Tailings Dewatering

Flotation tailings will be dewatered in a dedicated thickener. Thickener overflow will gravitate to the ACO process water dam. Thickener underflow, at 60% solids, will be pumped to the tailings pump hopper which also receives the Gold Plant tailings.

17.3.2.5 Regrind Circuit

Rougher flotation concentrate will be combined with secondary cleaner tailings and regrind mill product. This will be pumped to the regrind cyclone cluster. The cyclone overflow will gravitate to the cleaner conditioning tank and the underflow will gravitate into the single fine grinding regrind mill. Regrind mill discharge recirculating back to the regrind cyclone cluster feed pumps for classification.

17.3.2.6 Cleaner Flotation Circuit

The cleaner circuit will consist of a first cleaner circuit and a concentrate second cleaner circuit (sometimes referred to as recleaners). All mainstream production critical pumps will have an installed hot standby.

The regrind mill product will gravitate to an agitated cleaner conditioning tank which will overflow to the cleaner flotation cells. PAX and MIBC will be dosed into this conditioning tank. The concentrate from these first cleaner cells will be pumped through a sampler to the second cleaner conditioning tank from where it will gravitate to the second cleaner flotation cells. Additional dosing points for PAX and MIBC will be provided to dose into the second cleaner conditioning tank if required.

The tailings from the first cleaner will be pumped through a sampler to the flotation by-product pre-leach thickener. Here it will be thickened with the rougher scavenger concentrate and pumped to the CIL for cyanide leaching to recover contained metal values.

The second cleaner concentrate will be pumped through a sampler to the final concentrate thickener and the second cleaner tailings will be pumped through a sampler to the regrind mill feed tank and recirculated.

17.3.2.7 Concentrate Dewatering Circuit

Copper concentrate will be dewatered in a thickener before being pumped to a filter press for solid-liquid separation. The filtered product will be conveyed to a point for packaging either directly into a container, or into tote bags. The packaged concentrate will be weighed and sampled for metal accounting before dispatch to a smelter.

17.3.2.8 Combined Tailings Disposal

The flotation tailings and detoxified CIL tailings will be combined in a single tank and will be pumped to the TSF.

17.3.2.9 Flotation Reagents

The following reagents are required for Flotation. Additional detail is provided per Section 17.3.3:

- PAX – solids dissolved in water and dosed via pump.
- MIBC – dosed directly via pump.
- Flocculant – solids dissolved in water and dosed via pump.
- Hydrated lime slurry will be available via an extension of the Gold Plant system should it be required for concentrate cleaning. However, this reagent is not anticipated to be required.

The following dedicated services are provided:

- High pressure service air.
- Low pressure air for flotation aeration.
- Raw water storage and reticulation.
- Fire water reticulation.
- Process water and reticulation.
- Potable water reticulation.

The opportunity to rationalize some of the above services with the Gold Plant are options that will be explored as design progresses.

17.3.3 Reagents

17.3.3.1 Oxygen (O_2)

O_2 will be generated on site producing a minimum of 90% purity via PSA or VSA plant.

17.3.3.2 Sodium Cyanide (NaCN)

Sodium cyanide will be delivered in Isotainers in solid form. Dissolution and unloading will be carried out simultaneously by pumping diluted solution from the mixing tank through the Isotainer and back to the mixing tank. Cyanide will be added to the CIP and Elution Circuits via a ring main.

17.3.3.3 Quick Lime

Quicklime will be stored in silos adjacent to the SAG mill feed conveyor. Deliveries to site will be transferred to the silo for holding. Quicklime will be dosed directly to the SAG mill feed conveyor from the silo to provide pH control of the gold circuit milling and CIL areas.

17.3.3.4 Hydrated Lime

Hydrated lime is required for pH control of the cyanide detoxification system. This is achieved by mixing hydrated lime as received in an agitated tank with water to prepare a slurry that is then dosed by a ring main.

17.3.3.5 Activated Carbon

Activated carbon will be used to adsorb the gold and silver from solution. Ready use bags will be stored in the desorption section and will be loaded into the circuit over the fines removal screen.

17.3.3.6 Sodium Metabisulfite (SMBS)

SMBS will be delivered in bulk bags and mixed in the SMBS mixing tank and subsequently transferred to the SMBS storage tank. This solution is pumped directly to the cyanide detoxification circuit via dosing pumps.

17.3.3.7 Copper Sulphate ($CuSO_4$)

Copper sulphate will be delivered as needed for makeup and mixed in the Cu sulphate mixing tank. The mixed solution reports into the storage tank, from which it is pumped directly to the cyanide detoxification circuit via dosing pump.

17.3.3.8 Hydrochloric Acid (HCl)

HCl will be used for removing calcium scale from the carbon prior to gold and silver elution stripping. HCl will be supplied in 1 m^3 tote boxes at 32% HCl. Totes will be stored for ready use in the desorption section. Acid will be diluted with water to make up a 3% solution at the acid wash column.

17.3.3.9 Sodium Hydroxide (NaOH)

Sodium hydroxide (caustic soda) will be delivered to the site in solid form in bulk bags and mixed in the caustic soda mixing tank. The caustic soda solution will be pumped to the desorption and electrowinning circuits.

17.3.3.10 Leachaid

Leachaid will be used as the oxidant in the intensive leaching process. The bags will be manually emptied into a dosing hopper, and the Leachaid will be dosed into the hopper using a vibrating feeder.

17.3.3.11 Potassium Amyl Xanthate (PAX)

PAX will be used as the primary collector for flotation. PAX will be supplied to site in solid form and mixed in a mixing tank. The solution will then be transferred to a holding tank. PAX will be pumped from the holding tank via dedicated dosing pumps to the various dosing points.

17.3.3.12 Methyl Isobutyl Carbinol (MIBC)

MIBC will be delivered as a solution in a bulk container. This container is offloaded into a holding tank, from which it is pumped directly to the various dosing points in the flotation circuit via dedicated dosing pumps.

17.3.3.13 Magnafloc 1011

Magnafloc 1011 will be used for flocculation in all the thickeners. A screw feeder will deliver the required amount of flocculant to the mixing tank through an eductor to wet the flocculant with water. Flocculant will be mixed to a maximum 0.20% strength flocculant mix. This will provide sufficient time for full hydration between mixing and usage. Each thickener will have a dedicated variable speed dosing pump, with in-stream dilution.

17.3.3.14 Other Reagents and Consumables

Other reagents and consumables used are:

- Borax, nitre, and silica for gold room fluxes.
- Diesel fuel for the elution heater, reactivation kiln and smelting furnace.
- Grinding media for the SAG mill, ball mill and the tower mill.
- Anti-scale for water treatment delivered via dosing pumps direct from the supplier packaging.

18 PROJECT INFRASTRUCTURE

The Project is a greenfield project that will have supporting infrastructure both on and off site. Existing facilities on site including an exploration camp, airstrip, and site roads.

18.1 On site Infrastructure

The on site facilities will include a security entrance, site access roads, mine haul roads, open pit mine and waste rock storage areas, processing plant, laboratory and associated shops and offices, fuel storage and delivery facility, a contractor owned and operated fuel oil generating facility, explosives storage facility, camp, administrative buildings, emergency treatment facility, shops, warehouses, an airstrip, and laydown yards. The on site project facilities will be supported by utilities and. services

The utilities and services will include potable water systems, water supply system, and firewater system. An on site landfill will be utilized. The site will include a sewage collection, treatment, and disposal system. Additionally, a full communications system including radio, satellite, fibre optic network and a regional mobile telephone tower and system will be constructed.

18.2 Site Water Management Facilities

The purpose of site WMS include:

- Manage the Wynamu River and protect mine facilities for events up to the 100-year 24-hour storm;
- Develop a wetland within the Wetland to retain all site water for de-sedimentation prior to release to the environment via the Puruni River (Figure 18-3);
- Divert non-contact water to the Wetland;
- Collect contact water in ditches and convey it to the Wetland; and
- Release water from the Wetland to the Puruni River.

The WMS include mine haul roads acting as WMS structures, contact and non-contact diversion ditches, the Wetland and culverts regulating the flow and impoundment of site water for sedimentation.

18.3 Tailings Storage Facility

The TSF, located on the northeast side of the mine property, will be organized and operated in the area encompassing Module 1 and Module 3 (developed by KCB (2014)). The TSF has been designed for a storage capacity of 107 Mt within an impoundment with a final nominal capacity of 156 Mt of slurry tailings (Figure 18-1).

Tailings will be confined by site topography and constructed saddle dams, with the typical section being compacted saprolite shells with a chimney drain to relieve the head from the pond in the centre of the final dam and conduct seepage through finger drains downstream.

Water balance estimates indicate excess water volumes (mainly due to precipitation) during operations of the tailings modules. Water management includes the use of diversion channels and discharge of excess water volumes to the environment through operating spillways designed for the PMF.

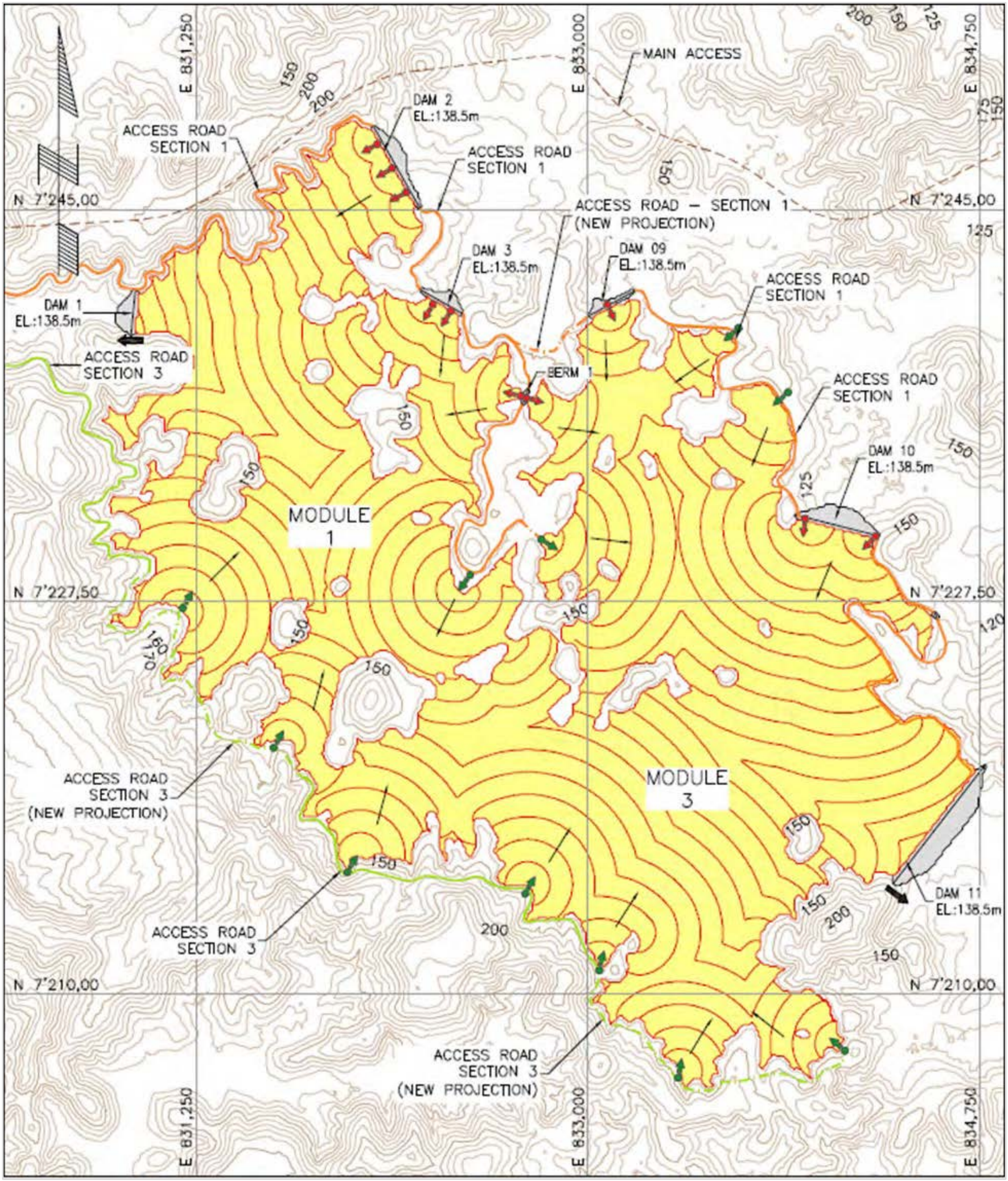


Figure 18-1: Ultimate condition – TSF storage

18.4 Off site Infrastructure

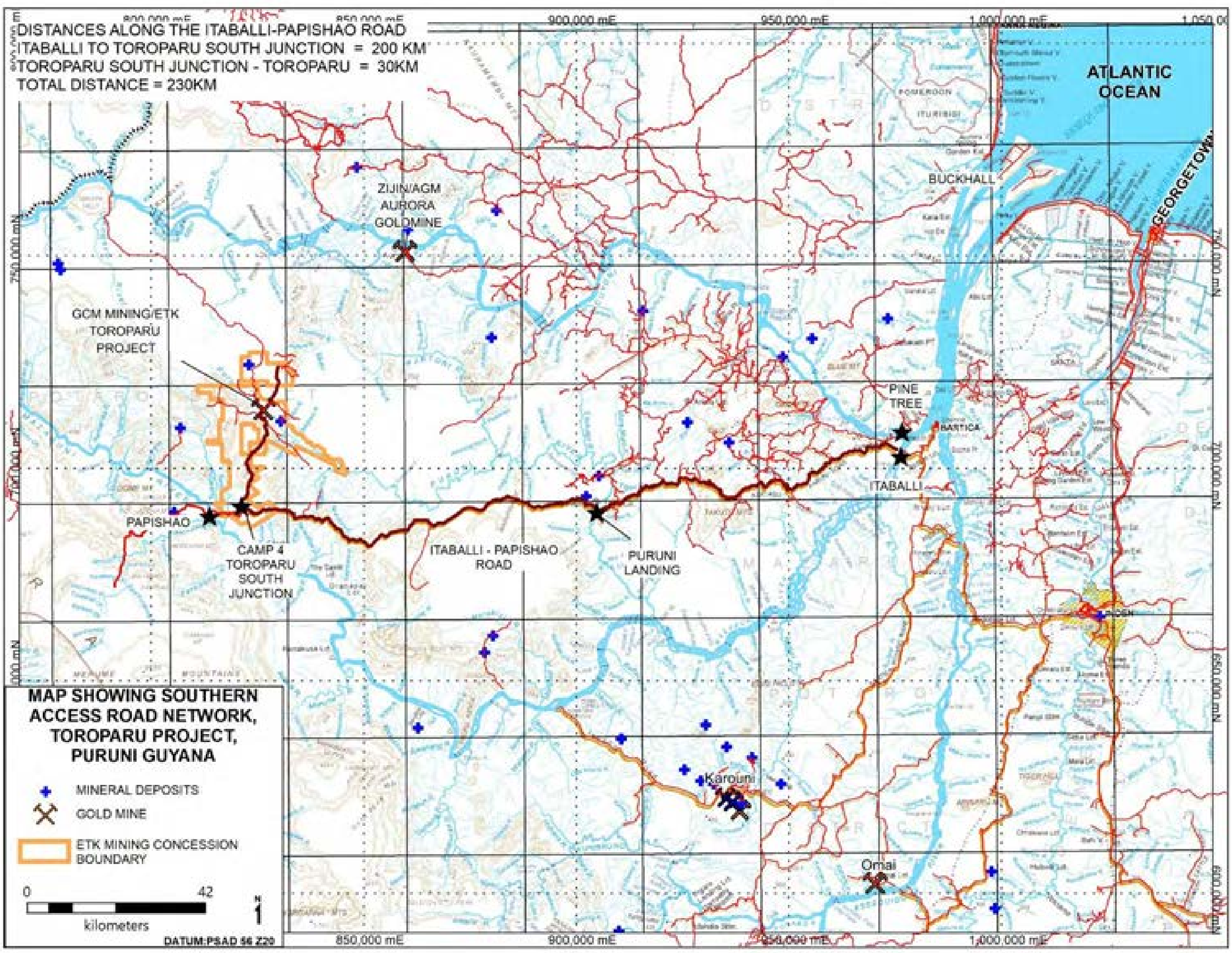
The off site facilities will include port access and access to the Project by road. The port facilities are located near Itaballi at a location on the south bank of the Cuyuni River approximately 2 miles upstream of the confluence with the Mazaruni River known as Pine Tree.

18.5 Infrastructure and Logistic Requirements

18.5.1 Project Logistics

During construction and mine operations, transportation of equipment, materials, and supplies will be delivered by barge and truck from Georgetown Harbor to a newly constructed port/wharf at Pine Tree and overland to the Project; and by air from Ogle International Airport in Georgetown.

Logistical infrastructure includes docking and transshipment facilities at third party ports in Georgetown, docking/unloading facilities for barges at the Pine Tree Port facility near Itiballi, overland access from Pine Tree to the Toroparu South Junction on the Itiballi-Puruni Landing-Papishao public road, then private road from the Toroparu South Junction to the Project site (Figure 18-2).



Source: ETK Mining, 2021

Figure 18-2: Project access

The existing 150-person exploration camp will serve as the primary base of operations for preliminary civil construction of permanent administration buildings, workshops, and the employee accommodations.

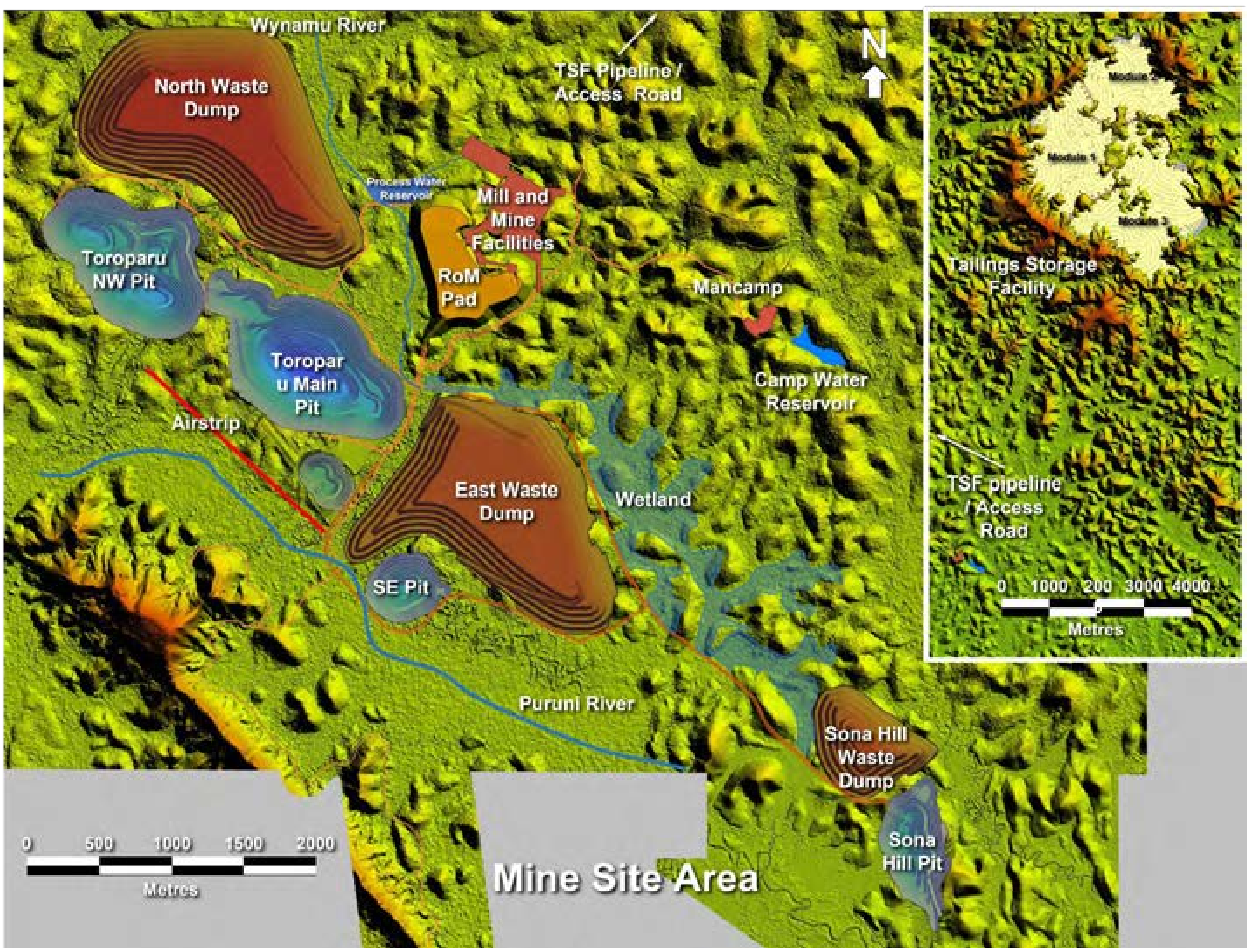
The existing 2,500 ft 24-hour flight certified Toroparu Airstrip provides access for domestic commercial and charter cargo and passenger aircraft operating form Ogle International Airport in Georgetown will be extended to 3,500 ft during preliminary civil construction works.

During production, the main access road will be used primarily for the supply of food, reagents, spare parts, mining supplies, and fuel, while the airstrip will be used for personnel transportation, light cargo, and emergency transportation.

18.5.2 On Site Infrastructure

The main Project site will contain five discreet open pits, waste rock stockpiles, a process facility with associated laboratory, reagent storage, and maintenance facilities; fuel oil fired power generation plant; mine maintenance workshops and open pit equipment yards.

Support facilities and structures include warehouse, office, change house facilities, explosives storage area, power generating station, fuel storage tanks, and a permanent accommodation complex (camp) located 1 km east and upwind of the processing facility. Figure 18-3 shows the general arrangement of the on site infrastructure.



Source: ETK Mining, 2021

Figure 18-3: Project on site infrastructure general arrangement drawing

The TSF is located approximately 10 km to the northeast of the process facility along the road to Toroparu Junction.

Service roads, mine and site water management dams, channels, berms, ponds, and ditches are all designed to handle site, process, and contact water. The criteria for selection of process and mine facility pads and other infrastructure near the mine pit include maintaining a safe elevation above the 100-year flood event boundary limit.

18.5.3 Site Roads

Site roads include haul roads suitable for use by mining trucks and service roads for use by smaller vehicles. Roads will be primarily constructed from weathered saprolitic rock obtained from pre-stripped saprolitic overburden waste from the open pits using cut and fill techniques to achieve design alignment and grades and compacted in small lifts to provide competent road foundations. Site road base will serve as diversions for surface water drainage in many locations. Haul and service roads will be covered with a layer of crushed rock sourced from project rock quarries during construction, and periodically from crushed fresh rock waste material sourced from the open pits to maintain all weather access during mining operations. In cases where mining operations share a road alignment with a service road, the flow of traffic will be separated with safety berms.

Site access roads which interconnect the various site services and areas area segregated to the maximum extent possible from the mine haul roads. Mine Haul Roads, which provide access to the pit, the waste rock dumps, and the ore stockpiles at the plant site will be constructed as WMS providing flood protection to various service areas, with strategically placed culverts to manage the flow of the Wynamu River, its tributaries, and site contact and non-contact water thru the mine site. The majority of the service roads are designed for two-way traffic with the maximum vehicle width of 6 m (2 x tractor trailer truck width).

The portion of the main access road linking the process facility to the TSF will be constructed as a single lane 5.5 m wide road within a 60 m wide cleared corridor with passing sections every 500 m. The road will be covered with lateritic rock from the access road laterite quarries or crushed rock from project rock quarries. The road from the process facility to the TSF will also be the corridor for the tailings and return water pipelines to and from the TSF.

18.5.4 Airstrip

The extended 1,750 m (5,740 ft) airstrip will accept cargo and passenger aircraft for several years. A second 1,750 m (5,740 ft.) airstrip will be constructed approximately 200 m (600 ft) toward the southeast as the open pits expand during the mine life (Figure 18-3).

Incoming and outgoing flights will be scheduled for daylight hours only. The existing airstrip at the Project is certified for nighttime use in case of medical emergencies. Aircraft maintenance and fuelling will be performed in Georgetown and no aviation fuel storage or maintenance facilities are required at site.

Figure 18-4 shows a photograph of the existing airstrip.

Source: ETK Mining, 2019

Figure 18-4: Existing airstrip photograph

18.5.5 Site Buildings and Facilities

18.5.5.1 Administration Buildings

The administration building will be located within the plant facilities area adjacent to the east of the process plant. The single story, pre-engineered, steel framed structure will be erected upon a spread footing foundation. The building will provide offices for the process operations staff, conference/training facilities, toilets, break room, and safety-security offices.

18.5.5.2 Process Warehouse, Workshop, Laboratory, and Storage Yard

A separate process operations workshop/warehouse with 1,800 m^2 of covered floor space will be constructed adjacent to the administration and fenced outdoor laydown area for equipment and bulk supplies. The laboratory building is a pre-engineered, steel framed single story, structure that will be located adjacent to the main process facilities. The laboratory will house sample preparation, assaying, testing facilities along with supporting sample and chemical storage rooms. The Process operations building contains the primary first aid and emergency treatment facility with a secondary facility located at the employee accommodations.

18.5.5.3 Mine Administration and Workshops

The mine administration and dry building is designed as a pre-engineered, steel framed single story structure that will house offices for the mine operations staff, change house facilities, conference/training facilities, toilets, break room, safety, and first aid.

The mine workshop building with, pre-engineered steel framed structure, will provide a maintenance area designed to repair and maintain the mine fleet and other mobile equipment including haul trucks, loaders, dozers, graders, etc. It will include several bays with an overhead crane for heavy mobile equipment that will be capable for maintaining the larger 785 trucks later in the mine life, and include repair and maintenance equipment, a machine shop, tire servicing area, and other work areas for other repairs. The mine area warehouse, electrical, and air compressor rooms will be housed in an adjacent building connected to the workshop. A separate truck wash station, equipped with a washing system with a water/oil separator for heavy mining equipment, will be installed outdoors.

18.5.5.4 Truck Fuel Facility and Equipment Ready Line

The vehicle fuelling facility and ready line will be located at the entrance to the plant area adjacent to the main haul road access to the mine pits. The fuelling facility will be operated by the independent fuel distributer in accordance with international standards providing sufficient fuel storage for minimum one month of operation as a buffer against disruption to the fuel supply chain.

18.5.5.5 Explosives Storage

The explosives magazine will be constructed and operated in accordance with Guyanese law. The storage magazines will allow for the bulk emulsion, blasting materials, and detonators to be stored separately. Explosives will be housed initially in an area located on high ground approximately 400 m north of the Puruni River and will be accessed via road from the site. The facility will be modular, so it can be relocated during the life of the mine as necessary. The site will be surrounded by a perimeter security fence with lights.

18.5.5.6 Employee Accommodations

The permanent employee accommodations will be located approximately 1,500 m east of the plant area on a rectangular level pad at 105 m elevation. The facility is upwind and up-gradient from the plant facility and is shielded from the plant area by several hilltop ridges and is designed as a dormitory style facility. The employee accommodation facility pad will allow capacity of more than 800 persons including dual occupancy, and single person rooms with shared and or private bath facilities, a full kitchen/dining hall, a recreation complex, a commercial laundry facility, a medical unit, an office unit that includes a greeting area and offices.

The employee accommodation facility will be managed by a third party provider that manages several large scale projects including for the Company operations in Colombia.

18.5.6 Power Supply and Distribution

18.5.6.1 Fuel Oil Power Plant

Power will be supplied thru a 20-year extendable power purchase agreement (PPA) between a third party independent power producer (IPP) operating a hybrid thermal-solar energy power plant. The Hybrid power plant will be financed, constructed, owned, and operated by the IPP during the life of the PPA. The power plant will initially consist of five 4.1 MW generators (initially four operating/one standby), based on an N+1 operational philosophy (with 100% power plant availability), for a total operating capacity of 16 MW and installed (N+1) capacity will be 16.4 MW. The plant will be expanded in 4 MW increments to meet the requirements of expanded process facility capacity (+ 14 MW in Year 6), underground mining (+ 4.5 MW in Year 11, plus minor capacity increases from open pit dewatering and other ancillary demands. The final configuration will consist of 9 (n+1) x 4.1 MW reciprocating engines and electrical generators. This final configuration will provide operating capacity of 33 MW a total installed capacity (N+1) of 37 MW.

Construction of the power plant will be thru a turnkey build, own, operate contract provided by the IPP including all buildings, cooling water management, maintenance and in plant switchgear. The power plant's generating sets will generate power at 13.8 kV, 60 Hz. The process facility's main electrical room will be fed with two 13.8 kV lines from the main power plant in order to ensure full redundancy.

Power distribution for the Project site will be by wooden pole overhead power lines from the substation at the generator plant routed to the process plant, mine support area and camp. Transformers and switchgear will be located at each of the buildings/facilities in the process plant with individual transformers and switchgear and motor control centres (MCC) for the local power loads. Typical voltages will be 4,160 volts for motors greater than 200 kW and at 13.2 kV for the larger ball mills. Additional transformers and electrical equipment will be provided for those electrical loads less than 200 kW at 480 volts.

Fuel and lubricants will be distributed under a long term supply agreement providing for delivery to the power plant and mine equipment from dedicated facilities owned and operated by the fuel distributor.

Power for the entry station will be provided by a standalone diesel generator to be utilized on an as needed basis.

18.5.7 Utilities and Services

18.5.7.1 Fresh and Process Water Supply, Fire Suppression and Distribution

Raw water required for the process operation, potable water, and fire suppression will be sourced from a freshwater pond formed from impounding a portion of the Wynamu River Flow upstream of the Process Mill, a surface water reservoir east of the employee accommodations, and rainwater harvesting systems.

The camp water reservoir (Figure 18-3) constructed east of the employee accommodations will supply fresh potable and fire suppression water for building services such as dining facilities, showers, and toilets. Potable water for the process facility, operations and maintenance, and employee accommodations facilities will be obtained from roof collection systems. Rainwater systems will be supplemented by treated water from the Water Reservoir. All potable water supply will be treated to meet World Health Organization (WHO) and Guyana EPA requirements.

Fresh raw water supply and process make up water for the Mill will be sourced from the Wynamu River Process Water Reservoir. This water will be used in the Au Room, heat exchange, cooling, reagent, laboratory, and other areas requiring clean water and stored in water storage tanks at the process plant.

Fire suppression water distribution covers all ancillary buildings, process plant, mine support facilities, workshops and yard equipment and will be sized to provide four hours of fire water demand and integrated with a fire alarm and detection systems.

18.5.7.2 Sewage Collection and Disposal

Sewage treatment facilities will be located downhill of the employee accommodations with buried sewer collection and transport to the sewage treatment facility. The sewage plant contains two independent containerized treatment lagoon systems working independently to provide redundancy during maintenance. The treated effluent will be released to the Wetland River via a feeder stream.

Sewage from the process facility, administration and maintenance facilities will be treated in a separate system with treated effluent discharged to the Wetland River via a local tributary.

18.5.7.3 Site Security

The principal site entry point from the access road will consist of a security entry building and vehicle access barrier (Main Gate). A masonry block gate house building will provide sanitary facilities, communications equipment and search facilities including metal detection. A weighbridge will be located adjacent to the gate house building to enable incoming and outgoing vehicle load monitoring. The existing Camp 4 Security Gate (Camp 4) located 25 km south of the Project at the Toroparu South Junction with the Itiballi-Puruni Landing-Papishao Road is the access point to the Upper Puruni Concession and the Project. Both Camp 4 and the Main Gate will be monitored by closed circuit television (CCTV) from the main security office located in the administration building. CCTV monitoring will also be installed at the process facility, gold room, power house, employee accommodations, and other critical locations.

18.5.7.4 Communications and IT Systems

The site will be connected to Guyana's GSM cellular network thru a 120 metre tower installed on Majuba Hill. This system will provide the primary communication for site including cellular phone service, high bandwidth data services, equipment location, and other networks within a 25 mile radius of the Project.

Point-to-point C-band satellite communication and back up Ku Band satellite systems will be maintained for redundancy. VHF/UHF radio communication is used as a backup system to Wi-Fi enabled satellite communication. A regional mobile phone tower installed at Papishao Landing is also accessible from the Project site.

Site UHF/VHF radio communication sets provide additional communication connectivity within a 10 km radius from the process facility. The site will be equipped with voice over internet protocol (VOIP), WAN ^ LAN Wi-Fi networks allowing fully integrated messaging, voice, and email communication across the site.

The IT system will be based at the operations and maintenance building and connected throughout the site by a fibre optic network connected to encrypted protocol independent multicasts (PIMS) and business networks through routers with firewalls and will provide remote access as required.

18.5.8 Site Preparation and Earthworks

18.5.8.1 Site Preparation

Before construction of the process, mining and camp facilities, areas within the construction limits will be cleared and grubbed of vegetation, rough graded, with fine grading of pads for buildings and facilities, and installation of drainage control structures.

18.5.8.2 Site Earthwork

Geotechnical site characterization east of the main pit (KCB, 2012) provides surficial and underlying geology information.

18.5.8.3 Clearing and Grubbing

The site area is approximately 99% covered by mixed forest, which contains both commercial and non-commercial trees. The clearing and grubbing operation will include the pad areas and a 10 m wide zone around the perimeter of the pad. Unmerchantable timber can be chopped. Stripped materials pushed with dozer and accommodated in local piles and burn. Merchantable timber logs should remain complete, accommodated in piles and later removed by timber contractor.

18.5.8.4 Stormwater Control

Stormwater features include perimeter ditches on the upstream side of the pads, a stormwater berm on the downstream side of the pads, and a stormwater pond sized to contain the 1,000-year storm events

18.5.8.5 Grading and Surfacing

Pads will be all weather surfaced with crushed gravel from the Project quarries placed over areas within each facility where foot and vehicle traffic are expected. The percentage of coverage varies at each facility pad depending on use.

18.5.8.6 Site Foundations

The 2012 KCB Geotechnical Site Characterization report describing geotechnical properties and subsurface conditions for the site indicate the upper subsurface profile generally consists of saprolitic soils of over 40 m depth into a transition zone and then to comparatively unweathered bedrock. Based on the geologic investigation completed by KCB, the upper subsurface material properties are considered competent in bearing and should not settle significantly under light loading.

The foundations for the administration buildings, warehouses, shops, and other support buildings consist of spread, strip, mat, or raft foundations depending on location, elevation, and proposed load. All shallow foundations are designed with a minimum Factor of Safety of 3.0 against bearing capacity failure. Deep foundations will be used to support heavy structures, or applications that have dynamic loads. These foundations will consist of helical piers bearing in the transition soils and competent bedrock.

18.6 Site Water Management

18.6.1 Site Water Management Structures

The purpose of the Site WMS is to:

- Manage the Wynamu River flow and protect mine facilities from flooding for events up to the 100-year 24-hour storm;
- Divert all non-contact water to the Wetland for single point discharge of settled water downstream of the mine site into the Puruni River; and
- Collect contact water in ditches and convey it to the Wetland.

The WMS components include:

- The North Waste Dump, Toroparu Mine Haul Roads & Culverts, RoM Pad and Channels to control the flow of the Wynamu River in its natural watershed thru the mine site.
- The East Waste Dump, SE, and Sona Hill Mine Haul Roads & Culverts, to divert the Wynamu River eastward into the Wetland to retain water for removal of sedimentation.
- Single spillway within the Sona Hill Haul Road system for monitoring discharge of all mine site water to the Puruni River and the environment.

Haul and Service Roads will be constructed on top levees with crests of 7 m to 10 m above the valley floor. These flood control structures are designed to the 1 in 100-year 24-hr peak flow elevation in the mine area and the Wynamu River with a minimum 0.5 m freeboard. The levees will be constructed with compacted saprolite, will have 2H:1 V side slopes and are designed to direct flows with no permanent ponding of water against them. The hydraulic section of the channel will be lined with rip rap for erosion protection. Geotextile, Rip Rap, and Veltivier Grass will be used for erosion control in other areas as necessary.

Levees will be constructed from compacted saprolite borrowed from pre-stripping of the Toroparu Main and NW Open Pit. Surfacing for the roads will be borrowed from a hardrock quarry crushing operation located at Majuba Hill.

Diversion ditches:

- Contact diversion ditches will collect run-off from disturbed areas around the mine facilities and divert them to the Wetland prior to discharge to the environment. Contact channels are located at the perimeter of the mine facilities and sized for operational stage conditions, for events up to the 1 in 10-year storm.
- Non-contact diversion ditches will collect run-off from beyond the mine and plant areas and divert directly to the environment to minimize the handling of contact water. The channels are located at the perimeter of the Main Pit, at the Crusher and Main Waste Dump areas.

Flood levees and berms:

- The levees are designed to protect the mine site from flooding from the Puruni River into the mine facilities. These flood control structures are designed to the 1 in 100-year 24-hr peak flow elevation in the mine area and the Puruni River with a minimum 0.5 m freeboard. The levees will be constructed with compacted saprolite, will have 2H:1 V side slopes and are designed to direct flows with no permanent ponding of water against them.
- Berms will be constructed to form a single barrier to separate the contact and non-contact ditches within the mine site. The berm crest elevation has been set to the greater of the contact or non-contact 1 in 100-year 24-hr peak flow. The berm side slopes have been set to 2H:1 V, based on no permanent ponding of water adjacent to them. They will be constructed with locally available saprolite.

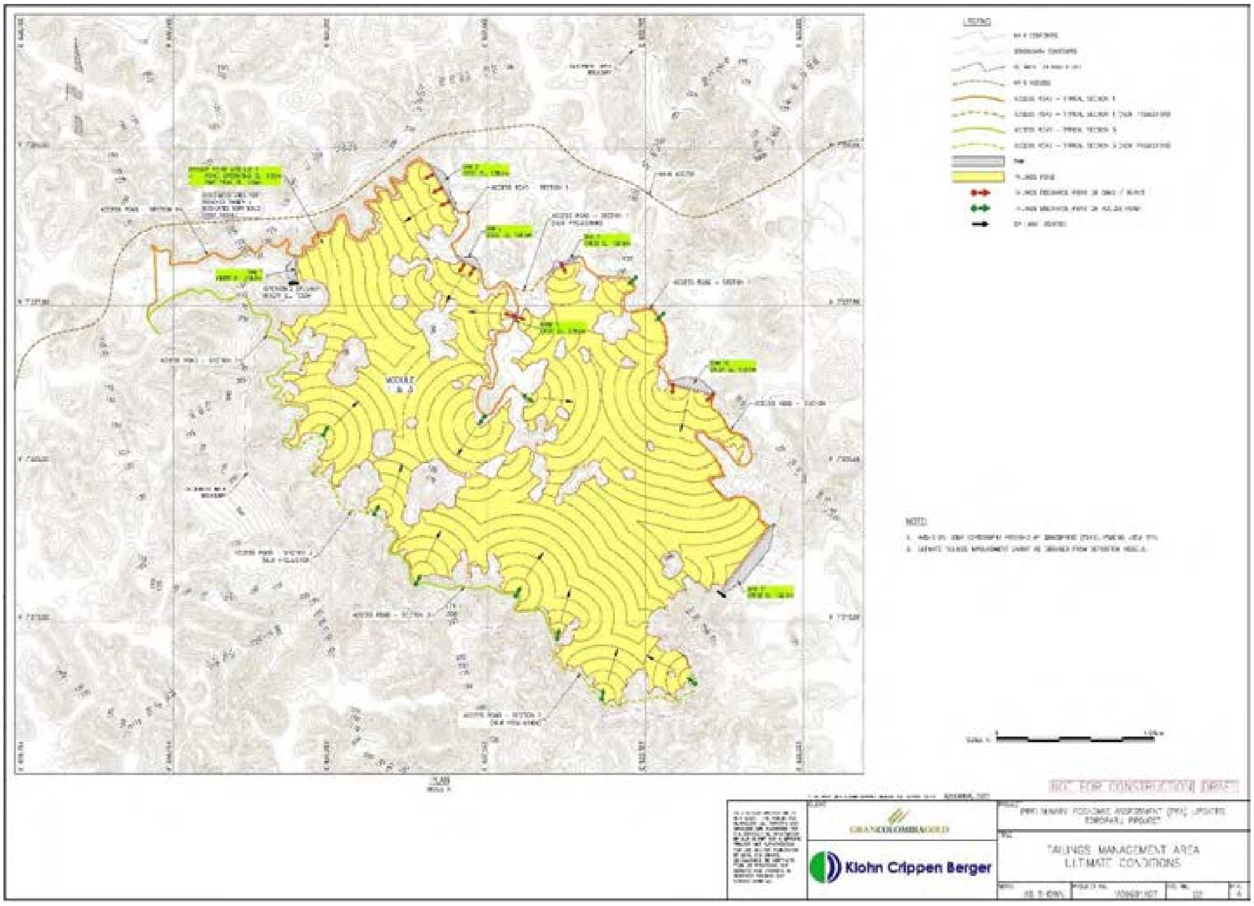
The Wetland will reduce flow sufficiently for settlement of suspended solids prior to discharge to the Puruni River thru a single spillway located in the Sona Hill Haul Road / Levees. The sediment ponds will comply with the operational objectives and the Method B sizing recommended in the Technical Guidance 7 (MOE 2015). The wetland is sized to:

- Allow settlement of the design settling particle (10 micron) for the 10-year 24-hour flood flow;
- Provide means to route the 200-year 24-hour storm event maintaining a minimum freeboard of 0.5 m;
- Provide adequate pond sizing for a minimum of 20-hour retention time for a 10-year 24-hour flood flow;
- Provide a minimum operational water depth of 1.0 m and a nominal allowance of 1.0 m for dead storage of sediment.

18.7 Tailings Storage Facility

The Tailings Storage Facility designs for the 2021 PEA are those defined in KCB feasibility level design report, Toroparu Project Tailings Management Area – Feasibility Design (KCB 2017) that defined the deposition of 133 million tonnes, expandable to 156 million tonne, capacity Tailings Management Area based on slurried (detoxed) CIL and flotation process tailings piped at a weighted average solids density of 54% to the TSF and discharged from the embankments and from selected locations along perimeter roads between modules and consolidation to a minimum dry density of 1.3 t/m^3 (Figure 18-3).

The TSF capacities and staging have been optimized for deposition of 107 million tonnes of detoxed CIL and flotation tailings produced from the PEA mine plan over the 24-year life of mine. The optimization is based on Modules 1 and 3 forming one large cell, with dams shared between both modules removed to reduce costs (Figure 18-5). A dry density value equal to 1 ton/m^3 is also considered based on the Tailings Laboratory Testing and Interpretation Program (KCB 2020) report.



Source: KCB, 2021

Figure 18-5: Location of berms and dams excluded in 221 PEA analysis

The TSF concept is optimized to store a total of 107.3 Mt and will be operated in three independent stages as outlined in Table 18-1.

Table 18-1: TSF Staging Summary

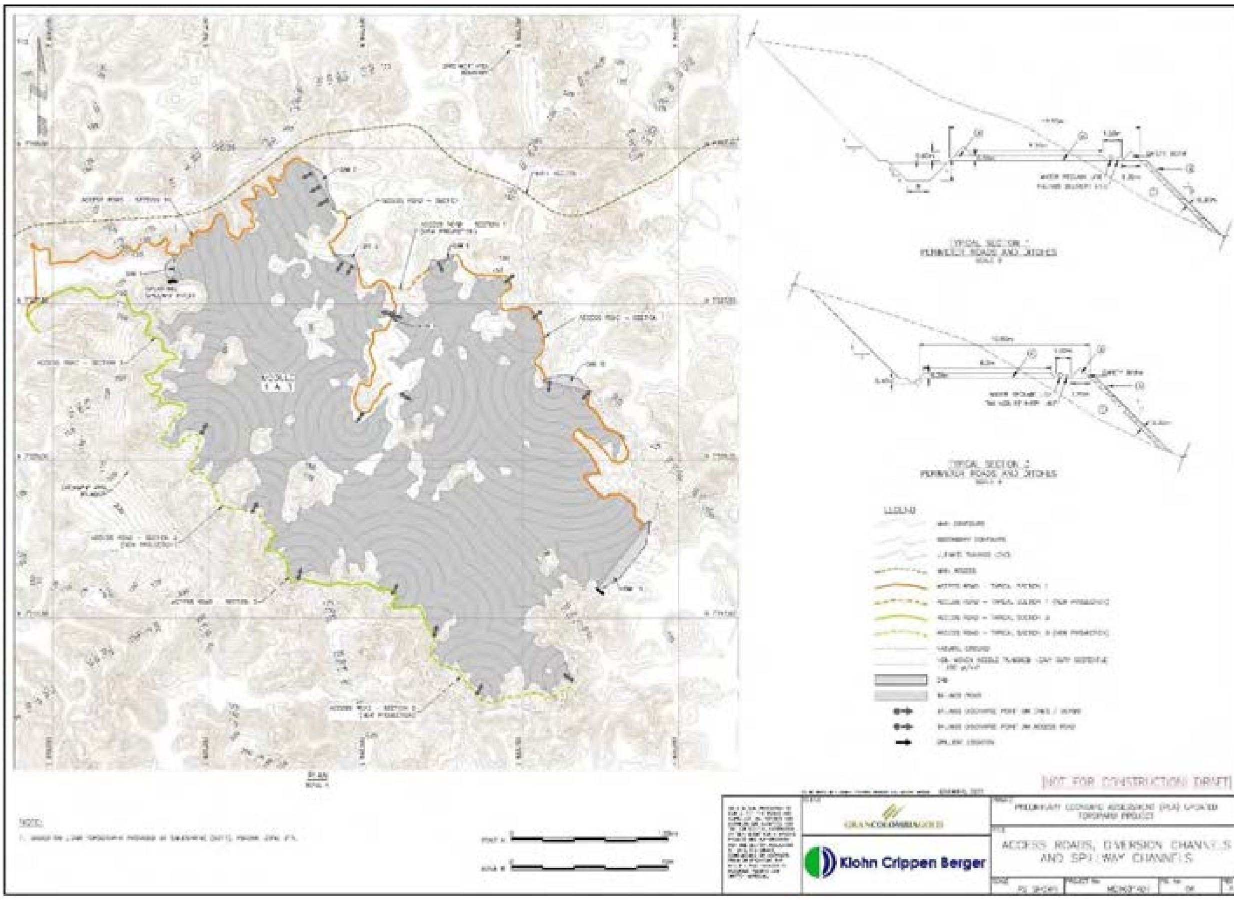
Stage	Modules Area Covered	Tailings Storage Capacity (Mt)	Period (Years)	MaximumDam Crest Elevation (m)	Dam Fill Volume (Mm^3)
Stage 1	Module 1	7.7	0 to 3	128.5	0.15
Stage 2	Module 1 & 3	30.7	4 to 10	131.5	0.60
Stage 3	Module 1 & 3	69.2	11 to 24	138.5	1.56
	Total:	107.6	24		2.31

Source: KCB, Nov 2021

18.7.1 Access Roads, Diversion Channels, and Spillways

Site preparation requirements for the dam footprints based on KCB (2017) and include the following:

- 10 km slurry tailings pipeline corridor / access road construction from plant site to tie-in point at TSF Module 1.
- 10.3 km of internal access road construction required for tailings spigots for tailings disposal in the Module 1 area and 9.8 km for the Module 3 area. Typical access and internal road sections based on KCB 2017 feasibility engineering (Figure 18-6).
- Dam 1 to 4 will be constructed in Stage 1 to isolate Module 3. Dam 4 and will function as a tailings overflow to fill Module 3 in Stage 2.

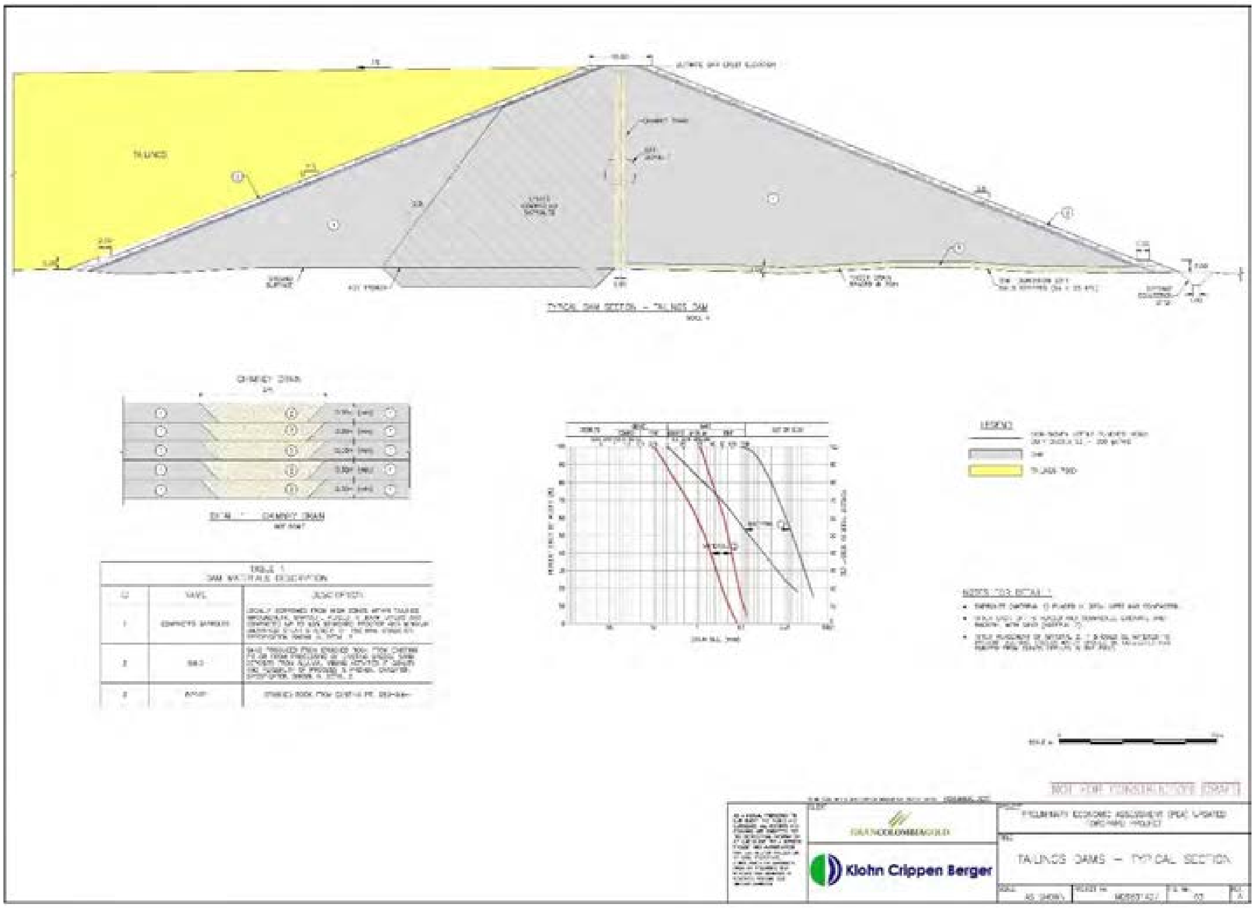


Source: KCB, Nov 2021

Figure 18-6: Access roads, diversion channels, and spillways

18.7.2 Dam Design

Dams are typical Brazilian design that uses the local saprolite material. Figure 18-7 shows the typical dam section proposed for the perimeter tailings retention dams (starter and ultimate dam).



Source: KCB, Nov 2021

Figure 18-7: Dam design

The dam is mostly compacted select saprolite, with a chimney drain to reduce the water level in the downstream shell and conduct seepage through sand finger drains to the downstream toe ditch.

The saprolite fill borrowed from adjacent hills and from excavation works during spillway construction will be placed and compacted during the dry season as a slightly sloping surface that sheds water and/or is covered with a thin film of pisolites. Internal drainage of the dams (chimney drain and finger drains) will be constructed with sand prepared from crushed rock from the construction quarry

Erosion of the upstream and downstream exposed surface slopes of the dams and embankments will be controlled with a layer of crushed rock (rip rap) on a heavy duty geotextile. This protective layer will be placed on the upstream and downstream slopes of these structures. This protection will be part of progressive and final closure works.

Construction sequence of the earth structures to be presented in each stage is described below:

- Stage 1: starts with the construction of Dams 1, 2, 3, and 4, reaching a uniform elevation of 128.5 masl. The perimeter access surrounding the module 1 area is developed at this stage.
- Stage 2: Dams 1, 2, and 3 will be raised to elevation 131.5 masl, Dams 10 and 11 will be constructed in the Module 3 area to elevation 125.6 masl. The Module 3 area will be filled in this stage through the overflow of Dam 4. For this stage, all perimeter accesses surrounding the Module 3 area will be developed.

- Stage 3: Dams 1, 2, 3, 10, and 11 are raised along with the construction of Dam 9 and Berm 1. All structures reach an elevation of 138.5 masl.

This design takes into consideration the results of the specific site investigation and geotechnical sampling program executed between February and March 2014, by KCB, the result of the site wide hydrology assessment (by KCB in 2014, and work done by others during earlier studies).

Predicted water quality of the decant pond meets the IFC discharge criteria. Water balance estimates indicate excess water volumes (mainly due to precipitation) during operations of the tailing's modules. Water management include use of diversion channels, reclaim of supernatant water volumes reclaimed to plant with a floating pump barges and discharge of excess water volumes to the environment through operating spillways designed for the PMF.

18.7.3 TSF Conceptual Closure Plan

The PEA configuration will allow the final closure of the entire cell contained in the disposal areas of Module 1 and Module 3.

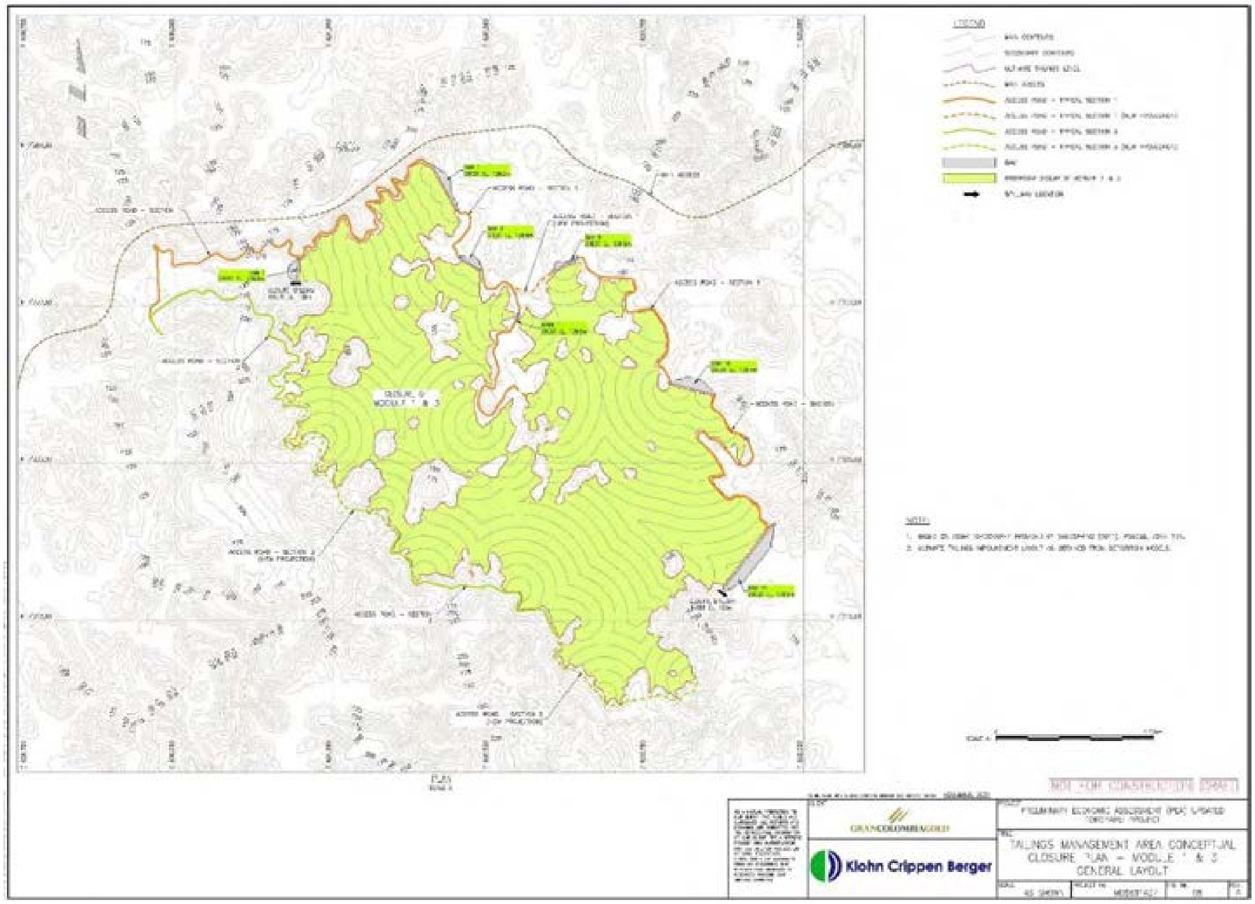
Any exposed tailings beach will be covered to reduce seepage. The closure cover could consist of a 1.0 m thick layer of saprolite under a 0.3 m thick layer of topsoil amended with fertilizer and seeded.

Drainage and water reclamation pipes will be removed. For final conditions, the weirs will be converted to closure weirs and will direct and discharge the PMF out of the TSF impoundments.

The proposed riprap layer on the downstream slopes of the dams will also serve as erosion control at closure. Post-closure monitoring and maintenance will be carried out for a minimum period of 5 years (or longer as required by Guyana EPA) including the following activities.

- Surface and groundwater quality inspection monthly during the first year of operation, and every 6 months thereafter (wet and dry seasons).
- Closure cover inspections to review vegetation progress and to identify any surface cracks or sloughing due to ongoing consolidation of the tailings beneath. Maintenance work should be carried out to repair any damage to closure covers.
- Inspection of the dams, including monitoring of slope movement, phreatic surface within the dam cover and identification of any erosion features.

The proposed closure is shown schematically in Figure 18-8. This closure plan is conceptual and will need to be updated in accordance with the proposed mine closure activities.



Source: KCB, Nov 2021

Figure 18-8: Conceptual closure plan

18.8 Off Site Infrastructure and Logistics Requirements

18.8.1 Access Road to Site

The Project has been accessed overland from Georgetown and from Tidewater thru Itaballi on the west bank of the Mazaruni River since 2003. Access to the Upper Puruni Property and the Project by road from Georgetown includes 128 km via paved highway south to Bartica, a ferry crossing of the Essequibo River at Bartica to Itaballi, then 200 km west on a public gravel road to the south gate at Toroparu Junction, and 25 km north to the Project site. Overland travel time is approximately 12 to 16 hours in the dry season (Figure 18-2).

Heavy equipment and cargo are transportable by small, ocean-going vessels and barges on the Essequibo River to Itaballi. There it is loaded on to trucks for the 230 km overland journey to Toroparu crossing the Puruni River at the town of Puruni Landing approximately 60 km from Itaballi on a GCM Mining-ETK operated 40 tonne ferry barge.

Upgrading and improving the 230 km (143 mi) roadway for construction and operation of Project includes:

- Brush back of vegetation along the 230 km alignment.
- Subgrade and Slope stabilization.
- Installation of culverts, small bridges, and the Puruni Landing Bridge.

- General grading and subgrade preparation earthwork.
- Fine grading, production, and placement of aggregate on slopes and laterite on roadway surfaces.

This access is being upgraded to serve as the primary access to the Project for construction and mining operations. Figure 18-2 shows the regional road systems and the overland access from Pine Tree to the Toroparu South Junction on the Itiballi-Puruni Landing-Papishao public road, then private road from the Toroparu South Junction to the Project site.

18.8.2 Port

The port facility at Pine Tree Landing will be a key feature in the mine supply chain. Pine Tree Landing is located on the right bank of the Cuyuni River and has historically been used by logging companies as a shipping point for lumber products and supplies.

Currently, Pine Tree Landing has approximately 225 m of waterfront, and is cleared of vegetation for approximately 500 m from the Cuyuni River, for a total area of approximately 32 acres. The existing facilities are very limited and will not be reutilized.

The proposed port facility will generally include the following items:

- wharf loading and discharge areas, logistics, truck maintenance and employee accommodations buildings, container, and equipment laydown areas, third party fuel storage and fuelling facility, power generation and related utilities.

Mine fuel supply logistics and infrastructure will be provided by an international fuel distributer (supplier) operating in Guyana, with delivery by barge into separate fuel oil and diesel storage tanks with capacity for one month of fuel for delivery in 19,000 L fuel trucks.

The port facility will accommodate ocean going barges which will transfer cargo between Georgetown and Pine Tree Landing via the Essequibo, Mazaruni and Cuyuni Rivers. Containerized cargo is anticipated to include both imported supplies and copper concentrate for export. Cargo at the port will be handled primarily with 40-ton forklifts and reach stackers. Heavy or oversized cargo will be handled by a mobile harbour crane or as roll-off cargo.

19 MARKET STUDIES AND CONTRACTS

The information contained in this report has been obtained from independent vendors or estimated from first principles based on the Project's local experience. The Project has been operating in Guyana since 1999.

19.1 Gold in Doré

All of the gold produced by the Project in the first 5 years of operation will be in the form of doré bars, with the actual amount depending on processing rates for each economic material in the year of production. Over the mine life, 80% of all Au produced will be produced in doré bars.

Au doré containing Ag will be shipped to refineries in North America or Europe for refining into refined Au bars meeting international specifications. The Company expects to be paid for the Au and Ag contained in the doré.

The market for Au bars is highly liquid, and the product is readily sold at spot Au prices to dealers, banks, or brokers directly from the refinery. As such, market studies, and entry strategies are not required for this product.

Metallurgical process studies confirm that doré will be produced at a specification similar to and at doré refining and transportation costs comparable with other operating Au mines in the region.

19.1.1 Gold and Copper in Concentrate

Starting in year 6 the balance of the Au, Cu and Ag will be recovered into a flotation Cu concentrate that will be transported for processing by a custom Cu smelter. The market for custom Cu concentrates is well developed.

Metallurgical process studies confirm that Cu concentrates will be produced at specifications acceptable to custom Cu smelters, and analytical results from those studies have been discussed with multinational Cu smelting companies who have provided preliminary acceptance of the concentrates and provided indicative terms for the treatment, refining, and delivery of these concentrates. These terms have been used by the Company in the financial analysis of the Project contained within this report (see Section 19.5).

19.2 Gold and Copper in Concentrate

Commodity prices used in the calculation of financial results presented in this PEA are US$1,500 per ounce or Au, US$20.22 per ounce of Ag and US$3.13 per pound for Cu. These prices reflect the long term price projection of the Company for gold, and the three-year (3-yr) trailing average price for Silver and Copper on October 30, 2021.

19.3 Contracts and Status

Currently there are no material contracts in place other than those disclosed in this document. It is anticipated that the following contracts will be in place upon Project commencement.

19.3.1 Metal Treatment, Refining and Transportation

- Modelled as an agreement for the secure transport of doré by air from the Project to refinery in North America or Europe;
- Modelled as an agreement for the refining of Au doré and delivery to the Company's designated bullion account;
- Modelled as an agreement for the treatment and refining of Cu concentrates;
- Modelled as an agreement for the transportation of containerized Cu concentrates from the designated concentrate shipping port to Georgetown, and transshipment for delivery to offshore custom smelter; and
- Modelled as an agreement for transportation and insurance for export of precious and base metal cargoes.

19.3.2 Supplier and Service Contracts

- Barge transportation of supplies between Georgetown Harbor and Pine Tree;
- Diesel and fuel oil supply and delivery to Pine Tree;
- Process reagents, consumables, and supply contracts;
- Equipment preventive maintenance services;
- Air transportation (Georgetown to site) services; and
- Site security services.

19.3.3 Potential Precious Metal Production Stream

This PEA considers a possible deal with Wheaton Precious Metals Corp. (Wheaton) where it purchases 10% of the Au production stream at US$400/oz payable Au and 50% of the Ag production stream at US$3.90/oz payable Ag. The acquisition cost of this precious metal production stream is estimated at US$138 million and is entirely included as a payment towards the initial capital. This acquisition cost was used to reduce the Project's capital requirements in the PEA economic model.

19.3.4 Indicative Terms

Terms used in the development of financial estimates of revenue and costs are as follows.

19.3.4.1 Doré Net Smelter Return

Based on actual costs from other Au producers with similar sized operations:

- Au Payable 99.95%;
- Ag Payable 99.25%;
- Au Refining Charge US$0.48/oz Au;
- Ag Refining Charge US$0.48/oz Ag; and
- Secured air transport and Insurance US$2.45/oz Doré.

19.3.4.2 Copper Concentrate Net Smelter Return

Based on indicative proposal received from multinational Cu company:

- Copper Payable Deduction of 1 percent point;

- Au Payable 97% of contained Au;
- Ag Payable 90% of contained Ag;
- Treatment Charge US$60.00/t;
- Copper Refining Charge US$0.060/payable lb Cu;
- Au Refining Charge US$4.50/ payable oz Au;
- Ag Refining Charge US$0.45/payable oz Ag;
- Penalties (Se / Bi) US$5.00 /t; and
- Conc. Transp. and Insurance 0.167% of payable metals.

19.3.4.3 Diesel and Fuel Oil Prices

Based on indicative pricing received from publicly traded multinational Fuel Distribution Company (FDC) with operations in the Caribbean and Guyana:

- Diesel Fuel (45-Cetane 0.5% S) US$0.780079/litre – cost, insurance, and freight (CIF) the Project; and
- Fuel Oil (IFO 180) US$0.6500/litre – CIF the Project.

Prices based on Mean Caribbean Price for referenced fuel delivered Free on Truck (FOT) the Project (includes delivery and storage in FDC owned tanks) averaged over three previous years to eliminate short-term price fluctuations.

Diesel fuel includes government excise tax of 15%. There are no excise taxes levied on fuel oil deliveries.

19.4 Royalties and Taxes

Royalties and taxes are governed by a mineral agreement (the "Mineral Agreement") executed between ETK Mining Ltd. and the Government of Guyana on November 9, 2011, which details all fiscal, property, import-export procedures, taxation provisions and other related conditions for the continued exploration, mine development and operation of the open pit and underground mine at the Project.

- Au Royalty 5% of Au sales at Au prices up to US$1,000/oz, 8% of Au sales at Au prices above US$1,000/oz;
- Ag Royalty 8% of Ag sales;
- Copper Royalty 1.5% of Cu sales;
- Corporate Tax Rate 30%; and
- Withholding Taxes None.
- Duties and VAT: Exempt on all imported equipment, supplies, and materials 16% on purchase of domestic sourced foods and supplies.

Additionally, royalty payments of $20,000,000 payable to the former surface owner at a rate of $2 million per year for 10 years following payback of all initial capital expenditures to ETK Mining are included in the PEA.

The $20 million payments are estimated at a gold price of $1,500 per ounce under an agreement "Surface Owner Royalty Agreement") executed between ETK Mining and the Alphonso & Sons on November 9, 2011.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL, OR COMMUNITY IMPACT

The following section discusses reasonably available information on environmental, permitting, and social or community factors related to the Project. For the most part, the information presented herein is predicated upon the 2019 PEA and no material changes have occurred in subsequent years or studies. Nordmin is of the opinion that the 2019 PEA information on environmental, permitting, and social or community factors related to the Project is adequate for the 2021 PEA presented herein.

The 2021 PEA project design reverts to the development of the off site infrastructure identified in the Company's 2013 PFS, specifically the River Port at Pine Tree Landing and the Itiballi-Puruni-Papishao Road, which is a public road in use since 2003. The Company's Environmental Authorization includes provisions for the improvements of these facilities.

The 2019 PEA project design included new and modified facilities to support a larger mining and processing operation and included a number of items which will require amendments or modifications to the existing applications and approvals outlined below.

- Modification of an existing river wharf, port and laydown operation on the Essequibo River.
- 47 km of new access road construction (linking to the existing Guyana Goldfields Aurora Mine).
- Construction of a new barge facility on the Cuyuni River.
- New and modified on-site access, service and haulage roads.
- Intermediate fuel oil (IFO) transport and depot facilities for the electric power generation facility.
- Entry station, operations man-camp, communications facility, potable water facility, and waste management facility.
- Modified mine dry and administration building, fuel depot, ready line, truck maintenance shop, warehouse facility and laydown area, and explosives storage facility.
- Modification to the TSF and Waste Rock Stockpile facilities.

20.1 Environmental Study Results

The initial environmental baseline studies were conducted in 2007, 2008 and 2010. The results were summarized, and the impacts were interpreted as part of the EIA submittal (ETK, 2012). Subsequent environmental studies included geochemical characterization by KCB and KCC Geoconsulting. The key environmental studies reviewed during the 2013 PFS included:

- A baseline report that summarized multiple periods of monitoring that included wet and dry seasons (both long and short seasons) and characterization of the site and regional vegetation, wildlife, topsoil, geology, surface water, groundwater, historic cultural properties, air quality and meteorological conditions;
- EIA (ETK, 2012);
- Geochemical characterization study, including static, leachate extraction, and laboratory kinetic tests, on representative drill core samples of the waste rock and low-grade economic material materials from the Toroparu and SE Pits (KCB, 2012a, b);

- Geochemistry analyses of the metallurgical test tailings generated from Toroparu and SE Pit economic material samples (KCB, 2011 and 2013); and
- Geochemical characterization study, including static, leachate extraction, and laboratory kinetic tests, on representative drill core samples of the waste rock and economic materials from the Sona Hill Pit (KCC 2019).

Baseline data on the physical environment and biodiversity were recorded and observed over an initial baseline period of May 2007 to May 2008. Baseline data were compiled to reflect conditions occurring in the four seasons over the period of record. Specifically, surveys were conducted in:

- Long Wet Season June-July 2007;
- Long Dry Season October 2007;
- Short Wet Season February 2008; and
- Short Dry season April 2008.

Another baseline characterization to gather data on physical environment and biodiversity was conducted in June – July of 2010 to supplement data initially recorded for the mine site. The objective was to expand the biodiversity baseline for the concession and to examine whether there were temporal differences in biodiversity data compiled over the preceding periods. The baseline investigations were all focused on the existing small scale open pit mine at the site and its immediate surroundings including the airstrip and site access road. Addendums to these baseline investigations will be necessary for the expansion areas of the proposed PEA mine plan, including, but not necessarily limited to the Sona Hill Pit and waste rock dump areas. It is not anticipated at this time that these areas will differ materially from those areas already studied.

20.1.1 Summary Results of Baseline Studies

The Project is located in northwestern Guyana, which has a tropical climate with two distinct wet and two distinct dry seasons. The initial baseline studies focused on biodiversity and water quality characterization. Since the Guyana EPA guidance does not include numeric water quality standards, IFC effluent requirements for mining operations were used for comparison purposes. The site vegetation is primarily secondary growth, mixed forest that shows indications of human disturbance. The mining concession is in a disturbed area consisting mainly of swamp, forests of leguminous trees called Morabukea, and mixed forests (ETK, 2010). The descriptions of the site conditions are summarized from the baseline studies document prepared by ETK (2010) and the EIA prepared by ETK (2012).

Surficial Soils

The baseline study included a geologic study and collection of surficial soil samples during two sampling events for analysis of constituents based on the Guyana EPA guidelines for mining. The geology of the area has already been described earlier in this report. It was noted that the baseline concentrations of metals are impacted by the remnant mine pit at the site. In addition, oil and grease detected in some samples were likely related to the ongoing mineral exploration activities.

The site, which is located within the South American tectonic plate, is in an area of low seismic activity. Five seismic events have been within recorded history within 300 km of the Project, with the highest magnitude reported as 4.90.

Climate

Guyana is located in the Equatorial Trough Zone (ETZ) and its weather and climate are influenced primarily by seasonal shifts of the ETZ and its associated rain-bands called the Inter Tropical Convergence Zone (ITCZ). Secondary influences on the climate are of Pacific origin. Formation of El Niño and La Niña can disturb the regular location of ITCZ and thus result in higher or lower than normal rainfall at specific locations. The El

Niño/La Niña is primarily responsible for inter-annual variations in rainfall. The entire area identified for development into the mine, like the rest of Guyana has a tropical climate and is not subject to extreme variations in temperature and humidity. The Company maintained a weather station at the Project during 2005 and during discrete periods of 2005. Very little climate data are available locally. During the 2005 monitoring the maximum annual precipitation was 2,100 mm, and temperatures ranged from a low of 20° to a high of 42° Celsius. The dry periods were January through March and August through October.

Winds were primarily from the north-northeast and the average maximum speed was 9.1 m/s. It is generally windier during the short, wet season.

It is recommended that monitoring at the weather station be re-established, and that the data collected include evaporation information. It would also be helpful to have precipitation data recorded at intervals less than one hour to understand the severity of storms.

Air Quality

In the area surrounding the Project site, there are no major industries that serve as significant sources of fixed or mobile atmospheric emissions. Aerial emissions are mostly attributable to the gases from rotting trees and other vegetative matter although some background emissions will inevitably be related to the operation of various small to medium sized motorized equipment in the area. Due to the high humidity and significant rainfall, dust levels on the roadways are generally low. Airborne discharges and particulate matter are not monitored in the area but are not expected to exceed the emission guidance established by the World Bank or WHO Ambient Air Quality guidelines.

Surface Water

The Project area is drained by the Puruni River and by several tributaries, the main one being the Wynamu River. The total estimated drainage area of the Puruni River is approximately 4,170 km^2. Approximately 375 km^2 of that drainage area is located upstream of the proposed mine site.

In 2010, water quality samples were obtained at three upstream locations to assess background water quality for the proposed mine. The water quality samples are demonstrative of water quality impacts from the former open pit mining operation and are also indicative of water quality prior to the commencement of additional mining operations. For comparison purposes the results were compared to the IFC effluent requirements for mining operations (2007), because the Guyana EPA guideline does not present permissible limits for water quality standards. The majority of the sample results were below the IFC effluent requirements; however, oil and grease and iron were exceeded in one or more samples.

Groundwater

The site occurs in the Precambrian crystalline basement rock section of Guyana. Five groundwater monitoring wells were installed to varying depths and at different locations during the first round of investigations conducted at the concession. The location of each well, the top of casing elevation and the static groundwater level, measured during each phase of baseline collection work at the site are detailed in Table 20-1.

Table 20-1: Monitor Well Locations and Groundwater Elevation Data

Well Identifier	Location (UTM)		Top of Casing Elevation (amsl)	Groundwater Elevation Measurement (amsl)				
	Easting	Northing		16/07/07	28/10/07	3/1/008	4/11/08	7/11/10
MW-1	824353	715217	32.87	31.70	31.04	32.03	30.48	32.20
MW-2	825149	713946	31.27	30.26	No data	31.14	28.33	31.14
MW-3	825948	713313	32.04	30.06	29.68	30.78	29.37	31.88
MW-4	826293	714540	32.50	30.01	29.73	30.63	29.14	31.14
MW-5	826854	713914	32.87	30.06	29.12	30.77	28.98	30.61

Source: ETK, 2012. amsl= above mean sea level

Groundwater levels recorded for the dry seasons are generally lower than those recorded for the wet seasons. This can be interpreted as being indicative of precipitation being the primary source of groundwater recharge in the Project area. No water was present in MW-3 during the second phase of the field work, coinciding with conditions recorded in the long dry season.

Very little data are available on groundwater flow parameters for that section of Guyana. Observations of remnant mines in the area indicate some groundwater inflow through the weathered unconsolidated material overlying intact rock. That flow may, however, be reflective of recharge by precipitation. Rising head insitu hydraulic conductivity tests were conducted in each monitor well after each baseline sampling event. The hydraulic conductivities ranged from 9.75×10-5 cm/sec in MW-3 to 7.43×10-8 cm/sec in MW-5.

Groundwater samples were collected and analyzed according to the parameters mandated by the Guyana EPA guidelines. The baseline results during the dry seasons were similar to those for the wet seasons. The Guyana EPA guideline does not present permissible limits for water quality standards, so for comparison purposes the results were compared to the IFC effluent requirements for mining operations (2007). Groundwater samples had slight exceedances of Iron and pH and very high exceedances of total suspended solids. Iron ranged from non-detectable at a detection limit of 0.03 mg/L to a high of 8.35 mg/L compared to the guidance value of 2.0 mg/L, while pH ranged from 5.62 to 8.91 in comparison to the guidance value of 6 to 9 pH s.u. Total suspended solids had very high exceedances ranging from 100 to 26,774 mg/L compared to a guidance value of 50 mg/L. Since naturally occurring groundwater typically would not exhibit high levels of total suspended solids, SRK recommends that the sampling methodology and water construction and development procedures be further reviewed to see if the well filter pack is appropriate, the well development was adequate, and the sampling technique is acceptable to international standards.

Archaeological Resources

The site is located in the Middle Mazaruni area. The mine site would have been encompassed by former Akawaio settlements and is contained within the Mazaruni Amerindian Reserve demarcated in 1904. This demarcation was reduced to the Upper Mazaruni Amerindian District in 1959. There are no records of any immigrant activity in the area.

The proposed site for the mine is located close to the source of the Puruni River, where some artisanal mining activities may have occurred in the area in the past. Flows in the Puruni at the proposed mine site are too low to have sustained year-long mining activity based on typical artisanal mining methods. The levels of mercury, which would have been used for beneficiation during Au mining, are not above detectable levels in soil, sediment or water samples recovered from the mine site vicinity. This serves to validate the assumption that the area may not have been extensively mined in the past. There are no remnants of mining activity in the area, except for the presence of the former open pit mine in the concession.

There is no evidence above ground surface of historical buildings or sites left over from settlements formerly in the area and there do not appear to be any heritage issues local to the proposed mine site.

Flora

The flora around the Project area was examined in 2007, 2008, and 2010. The floral survey consisted of both a forest inventory and a floristic survey. Four of the species identified in the area are non-commercial species. Total standing volume of timber trees estimated to be present is approximately 5,913,000 m3. The four most abundant commercial timber species recorded in the concession area are Mora gonggrijpii, Mora excelsa, Eshwileria sagotiana, and Clathrotropis macrocarpa. Two of these are included among the most harvested timber species locally.

In general, tropical rainforests are characterized by high tree diversities; the floral survey recorded a total of 55 plants species comprised of 38 timber tree species and 17 lesser plant species. The area can therefore be described as having low species diversity. Because the Project area is in a secondary growth forest, slow growing species in disturbed areas will fail to recover after some amount of disturbance. This may ultimately lead to a reduction in the number and diversity of species.

Terrestrial Fauna

A total of 19 mammalian species were found in the Project area. The majority of these species are fairly common in Guyana. However, the Panthera onca, Oryzoborus angolensis, and the Tapirus terrestris have special classification by the Convention on International Trade in Endangered Species of Wild Fauna and Flora (CITES), 2013 and the International Union for Conservation of Nature (IUCN), 2010. The surveys concluded that although there are high numbers of bats, only a few species are represented in the area.

A total of 52 individual fish species were recorded during the 2007, 2008, and 2010 aquatic surveys of the Project site and expanded footprint that took place in the Puruni river and Wynamu River. The majority of the listed species are found in the highlands of the Guyana Shield, especially in the Potaro river basin, or in the Rupununi Savannas Region. Review of this list against the fish species recorded at the Project revealed no Guyana endemics.

Several fish species recorded in the expanded Project footprint have economic and social values. The Erythrinids, including Haimara (Hoplias aimara), Huri (Hoplias malabaricus), and Yarrow (Hoplerythrinus unitaeniatus) are important food sources.

Avifauna

No endangered species were encountered, but three bird species: Ara chloroptera, Pionites melanocephala, and Pionus menstruus are listed in CITES.

Herpetofauna

The relative species abundance determined during the survey indicates that the habitat may be ideal for amphibians and reptiles. None of the species documented are endemic species or listed by CITES.

Fauna Species of Interest

No locally rare, threatened, or endangered species were recorded during the surveys; however, a number of species identified are listed by CITES. These along with other species listed by CITES and their status are provided in Table 20-2.

Table 20-2: International Status of Species

Species	Common Name	International Status	Local Status (Unofficial)
Caiman crocodiles	Spectacle Caiman	CITES Appendix II/III	Fairly Common
Paleosuchus sp	Dwarf Caiman	CITES Appendix II/III	Common
Eunectes murinus	Anaconda	CITES Appendix II/III	Common
Epipedobates trivittatus	Poison Frog	CITES Appendix II/III	Uncommon
Epipedobates sp	Poison Frog	CITES Appendix II/III	Uncommon
Panthera onca	Jaguar	CITES Appendix I, IUCN Lower Risk – Near Threatened Species	Uncommon
Tapirus terrestris	Lowland (Brazilian) tapir	IUCN Lower Risk – Near Threatened Species	Uncommon
Cebus olivaceus	Wedged-capped Monkey	CITES Appendix II/III	Common
Saimiri scuries	Squirrel Monkey	CITES Appendix II/III	Common
Agouti paca	Labba	CITES Appendix II/III	Common
Oryzoborus angolensis	Lesser seed Finch	CITES Appendix I	Uncommon
Amazona amazonica	Orange-winged parrot	CITES Appendix II/III	Uncommon

Species	Common Name	International Status	Local Status (Unofficial)
Amazona farinosa	Mealy Parrot	CITES Appendix II/III	Fairly Common
Amazona ochrophala	Yellow-crowned Parrot	CITES Appendix II/III	Fairly Common
Ara chloropterus	Red and Green Macaw	CITES Appendix II/III	Common
Brotogeris chrysoptera	Golden Winged Parakeets	CITES Appendix II/III	Common
Deroptyus accipitrinus	Red-fan Parrot	CITES Appendix II/III	Common
Piontes melanocephalus	Black-headed Parrot	CITES Appendix II/III	Common
Pionus fuscus	Dusky Parrot	CITES Appendix II/III	Common
Pionus menstruus	Blue head Parrot	CITES Appendix II/III	Common
Pyrrhura picta	Painted Parakeets	CITES Appendix II/III	Common
Pteroglossus aracari	Black-necked Aracari	CITES Appendix II/III	Common

Source: ETK, 2012.

Classification includes three CITES appendices:

- Appendix I includes species threatened with extinction.
- Appendix II includes species not necessarily threatened with extinction, but in which trade must be controlled in order to avoid utilization incompatible with their survival.
- Appendix III contains species that are protected in at least one country.

20.1.2 Geochemical Studies: Toroparu and SE Pit Tailings, Waste Rock and Low-Grade Economic Material

The exposure of tailings, waste rock, open pit walls, the LGO stockpile, and tailings to air and water may result in the generation of ARD and metal leaching (ML). Geochemical characterization studies were conducted for the Toroparu and SE Pits by KCB from 2011 to 2013 (KCB, 2011, 2012, 2012a, and 2013) on the dominant bedrock lithologies representing waste rock and LGO, and metallurgical tailings representing the three main economic material types (saprolite, ACO and LCO) for ARD/ML potential. The geochemical characterization studies were based on industry guidance documents (International Network for Acid Prevention [INAP], 2013 and Price, 2009). Lithologies identified as potential waste rock and LGO included Acid Intrusive, Fragmental Mafic Volcanic, Granodiorite; Mixed Facies, saprolite, and Undifferentiated Intermediate Volcanics.

The original waste rock, LGO and tailings samples were analyzed for static testing including mineralogical analysis, solid-phase elemental analysis and ABA. In addition, the waste rock and LGO samples were analyzed for leachate extraction tests consisting of Shake Flask Extraction and NAG tests. The tailings samples were also subjected to supernatant aging tests.

The leachate extraction and laboratory kinetic test analytical tests results were compared for screening purposes only to the IFC mine effluents guidelines (2007), the Canadian (federal) (2012) water quality guidelines, and the British Columbia (provincial) water quality guidelines (2012) for the protection of freshwater aquatic life, as no Guyanese concentration limits have been established for effluent discharge or for the aquatic receiving environment (e.g., a river).

Waste Rock and Low-Grade Economic Material Geochemistry Testing and Results

The first phase of waste rock and LGO geochemical testing included 150 waste grade core samples that were selected from the major lithologies for static testing. However, these samples were selected based on the previous 2011 mine plan. Based on the current resource and pit design, the number of drill core samples that are within the pit shell is reduced from 150 to 62 samples. The samples were selected from the DDH core and/or assay reject materials. The sample selection was based on the understanding of the major lithology types, their distribution throughout the deposit, the economic material cut-off grades, pit geometry and assay data in the DDH database, based on the 2011 mine plan. The waste rock cut-off used was 0.3 AuEq (g/t). The lithologies and their representative proportions are as follows: Acid Intrusive (6%), Fragmental Mafic Volcanic (10%), Granodiorite (<1.0%), Calcareous Intermediate Dyke (2%), Mixed Facies (55.0%); massive intermediate volcanic (28%); porphyritic intermediate volcanic (6.0%) saprolite (9%); Transition Zone (2%), and Undifferentiated Intermediate Volcanics (4%).

The analytical program consisted of the following static tests:

- Solid-phase elemental analysis using aqua regia digestion followed by Inductively Coupled Plasma Mass Spectrometry;
- Fizz test;
- Paste pH;
- Sulphur speciation;
- Total Inorganic carbon as CO2; and
- Modified Sobek Neutralization Potential.

All analyses were carried out by Maxxam Analytics located in Burnaby, British Columbia.

For the second phase of geochemical testing, a sub-set of 11 samples were selected for further leachate extraction testing and mineralogical analyses (KCB, 2013). All samples are considered to be waste rock based on assay results less than 0.2 g/t Au and represented six different lithologies (Acid Intrusive, Fragmental Mafic Volcanic, Granodiorite, Mixed Facies, saprolite and Undifferentiated Intermediate Volcanics). The analytical program consisted of the following leachate extraction tests:

- Mineralogy by Optical Petrography and X-ray Diffraction with Rietveld-refinement;
- NAG tests; and
- Shake Flask Extraction tests.

Optical petrography was completed by Mineral Services Inc. located in North Vancouver, Canada. The X-ray Diffraction with Rietveld-refinement analysis was carried out by the Department of Earth and Ocean Sciences, University of British Columbia, Vancouver, Canada. All other analysis and test work were carried out by Maxxam Analytics located in Burnaby, British Columbia, Canada.

Metal Leaching Risk of Waste Rock and LGO

Results of the solid-phase elemental analysis indicated that the lithologies included high concentrations of silver, arsenic, cobalt, chromium, copper, nickel, molybdenum, sulphur, and selenium in comparison to average crustal abundance of high-calcium granite. There was a wide variation between the different lithologic units. The LGO samples indicated high solid-phase concentrations of silver, copper, nickel, selenium, and sulphur. Therefore, the elevated concentrations of these elements in the solid-phase may be at risk of leaching under site-specific field conditions.

The results of Shake Flask Extraction tests indicated elevated leachate concentrations of aluminum and selenium, relative to water quality guidelines, from non-saprolite waste rock lithologies and LGO samples. The more aggressive NAG tests also indicated elevated chromium and copper in leachate from one or more samples, relative to water quality guidelines. For the saprolite waste rock samples, phosphorous, relative to water quality guidelines, was leached and readily soluble from Shake Flask Extraction tests. For the more aggressive NAG test, chromium and silver were also leached and readily soluble. With the exception of phosphorous, these leachate extraction test results are consistent with elevated solid-phase concentrations of silver and chromium.

The NAG extraction test results reported concentrations of phosphorus, aluminum, chromium, copper, selenium, and silver were elevated above reference guidance using Canadian Council of Ministers of the Environment water quality guidelines for the protection of aquatic life. The pH values for waste rock samples were above the guidance values, except for one saprolite sample that had a pH below the guidance value of 6.5 s.u. The Shake Flask Extraction results indicated that phosphorus, aluminum, and selenium concentrations were elevated, and that one sample had a low pH.

There are elevated concentrations of silver, arsenic, cobalt, copper, chromium, molybdenum, nickel, selenium, and sulphur compared to crustal rocks. The short-term leaching tests reported leaching of aluminum, selenium, chromium, and copper from non-saprolite waste rock. The saprolite waste rock was observed to leach phosphorus, chromium, and silver at concentrations above the aquatic life guidelines.

Acid Rock Drainage Risk of Waste Rock and LGO

The paste pH results indicated that the major lithologies are alkaline with the exception of the saprolite and the Transition Zone samples. The saprolite samples were slightly acidic to neutral while the transition zone samples were neutral to alkaline. These results indicate that no acidity was released from any of the samples except from the saprolite samples. The alkaline results indicate effective carbonate buffering. The LGO samples are alkaline, which indicates a potential buffering capacity.

The waste rock sample results were low in total sulphur and sulphide-sulphur content, and the associated calculated sulphide-based Acid Potential (AP) values were also low. The LGO total sulphur and sulphide-sulphur contents and sulphide-based AP were also low.

The waste rock lithologies and LGO samples contained low to moderate Neutralization Potential, (NP) with the exception of the saprolite and Transition Zone samples, which contained negligible NP. With the exception of the saprolite samples, the Net Potential Ratio (NPR), the ratio of NP to AP, calculated for the waste rock lithologies indicated that the waste rock was classified as NPAG, and therefore have a very low potential to generate ARD. The Transition Zone samples also had a low ARD potential. For the saprolite, most of the samples were Acid Generating (AG), although some were classified as PAG and NPAG. The LGO samples were NPAG.

The NAG pH results confirmed the not-PAG ARD risk of waste rock and LGO samples. The saprolite samples had the lowest pH results (5.9 and 6.8 pH s.u.) whereas the other samples had NAG pH results between 11.0 and 11.5 pH s.u. A NAG pH of 4.5 s.u. or less is indicative of PAG material. Based on the results of the NAG tests, most of the samples had very low sulphide content and an abundance of neutralizing minerals. The saprolite samples had mixed results regarding its potential to produce acid, but it was concluded to be most likely NPAG due to its low sulphide-sulphur content (0.07% wt.).

Humidity cell testing was recommended to be completed to further assess metal leaching of waste rock, LGO and open pit walls under alkaline conditions.

Tailings Geochemistry Testing and Results

Static testing was completed on six tailings samples and four supernatant samples by KCB (2013). Three material types (saprolite, ACO and LCO) were subjected to gravity separation, flotation and cyanide leaching to create tailings of each type. The six tailings samples in the geochemical testing included the following:

- Two samples of ACO cleaner detoxified tailing pulp combined with rougher tailings;
- One LCO rougher and cleaner composite tailings sample; and
- Three saprolite samples (cleaner detoxified slurry, coarse cyanide leach slurry and coarse flotation slurry).

The samples of these three metallurgical tailings streams (saprolite, ACO and LCO) were submitted for geochemical characterization. The tailings samples were submitted for the following static and leachate extraction tests:

- Mineralogical analysis;
- Solid-phase elemental analysis;
- ABA; and
- Supernatant aging tests.

All analysis and test work were carried out by Maxxam Analytics located in Burnaby, British Columbia, Canada.

Metal Leaching Risk of Tailings

A screening level comparison of the elemental analysis of the tailings solids to three times average crustal abundances indicated that silver, arsenic, bismuth, cadmium, cobalt, chromium, copper, molybdenum, lead, antimony, tin, tungsten zinc was elevated, and may be at risk of leaching under site-specific field conditions.

A total of four tailings slurry samples, including two ACO, one LCO and one saprolite tailings slurry sample following detoxification, were selected for an aging test of the supernatant to provide an indication of how the quality of the TMA ponded water may vary over time. The aging tests were conducted for a period of 90 days, with supernatant sampling analysis at 1, 7, 14, 21, 20, 60, and 90 days.

The tailings supernatant aging test results indicated that fluoride, nitrite, ammonium, CNWAD, aluminum, arsenic, chromium, cobalt, copper, iron, molybdenum, and selenium concentrations were above the Canadian and British Columbia water quality guidelines and therefore may be parameters of environmental concern (KCB, 2013). Additionally, iron and CNWAD were elevated in LCO and/or ACO tailings supernatant.

The TSF will receive a combination of precipitation, water treatment plant brine, and supernatant from the tailings slurry. Although the metallurgical tailings leachate extraction test results indicated elevated concentrations that may be soluble and mobile under laboratory test conditions, the results do not imply that they will be elevated to levels above these guidelines under site-specific field conditions, rather they identify elements that are prone to leaching. The TSF will receive a combination of precipitation, water treatment plant brine, and supernatant from the tailings slurry. The TSF water quality will be influenced by contributions from all these sources. The TSF design assumes that the natural low permeability of the surficial soils, and the lower concentrations of elements in the TSF pond due to attenuation from natural degradation, settling, and mixing with precipitation, which averages about 2.6 m annually, will reduce concentrations in any TSF discharge effluent to the aquatic receiving environment. Additional analysis (i.e., predictive water quality modelling) will be needed in a later phase to verify this assumption.

Additionally, the TSF management strategy of subaqueous tailings deposition combined with a cyanide destruction goal of 0.5 mg/L may be sufficient to mitigate potential environmental impacts to the aquatic receiving environment from effluent discharge from the TSF (KCB, 2013). Kinetic testing was recommended to evaluate the behaviour of the tailings under flooded conditions.

Acid Rock Drainage Risk of Tailings

The ABA results indicated that all samples have a neutral to alkaline paste pH and an acid-buffering capacity. However, the neutral to alkaline paste pH values are expected for metallurgical testing with lime addition.

The total sulphur ranged from below the detection limit to 0.08% wt. Sulphur speciation analyses indicated very low to negligible sulphide-sulphur concentrations with only one sample result reported above the detection limit of 0.03% wt. The sulphide-based AP of all the tailings samples varied between 0.16 kg and 0.94 kg CaCO3/t, with a median value of 0.23 kg CaCO3/t, indicating a low sulphide reservoir to oxidize and generate acidity. The inorganic carbon measured as CO2 varied between 0.010% and 2.49%. The corresponding Inorganic Carbon Neutralization Potential (Inorg-CaNP) ranged from 8.33 kg to 207.5 kg CaCO3/t, with a median value of 82.5 kg CaCO3/t.

The Sobek NP ranged from 10 kg to 134 kg CaCO3/t, with a median value of 56.9 kg CaCO3/t. The bulk of the Neutralization Potential appears to be from reactive carbonates and/or the addition of lime during the metallurgical testing. The saprolite tailings samples contained moderate NP (7 kg to 10 kg CaCO3/t) and the two Cu bearing material tailings contained high NP (104 kg to 134 kg CaCO3/t).

The tailings samples were classified as NPAG based on the sulphide-sulphur values and are therefore considered to have negligible risk of ARD (KCB, 2013).

20.1.3 Geochemical Studies: Sona Hill Pit Waste Rock and Economic Material

Geochemical characterization studies were conducted for the Sona Hill Pit by KCC Geoconsulting from 2018 to 2019 (KCC, 2018 and 2019) on the dominant bedrock lithologies/rock types representing waste rock and economic material for ARD/ML potential. The geochemical characterization studies were based on industry guidance documents (International Network for Acid Prevention (INAP), 2013 and Price, 2009). The following ten (10) dominant lithologies/rock types were characterized: Cataclastic Undifferentiated Hydrothermal Facies/Cataclastic Undifferentiated Hydrothermal Facies (CATHYDR/CHYDR), Dyke/Dyke (DYKE/DYKE), Undifferentiated Hydrothermal Facies/Undifferentiated Hydrothermal Facies (HYDR/HYDR), Intrusive/Granodiorite (INTRUS/GRDT), Intrusive/Undifferentiated Intermediate Intrusive (INTRUS/IINT), Intrusive/Quartz Diorite (INTRUS/DIOTQ), Intrusive/Undifferentiated Acid Intrusive (INTRUS/IACI), Undifferentiated Acid Volcanics/ Undifferentiated Acid Volcanics (VACI/VACI), Volcanics/Undifferentiated Intermediate Volcanics (VOLC/VINT), Quartz Vein/Quartz Vein Parallel (QZVN/QZP), Quartz Vein/Quartz Vein Oblique (QZVN/QZO), Tectonic Breccia/Tectonic Breccia (TCBC/TCBC), Weathering Zone/Saprolite (WEATH/SAP, SAPR, APRK), Weathering Zone/Saprolite-Transition (WEATH/SPRK-TZ), Weathering Zone/Saprolite Undifferentiated Intermediate Intrusive (WEATH/SPIINT), and Weathering Zone/Saprolite Undifferentiated Volcanic Sediment ((WEATH/SPVSCD).

The static testing completed as part of the geochemistry characterization program was undertaken on sixty-nine (69) drill core samples representing waste rock and four (4) drill core samples representing economic material. A sub-set of seven (7) samples were selected for leachate extraction tests, and three (3) samples were selected for laboratory kinetic tests, that included particle size and mineralogical analyses.

The geochemical analyses were the same as was completed on the Toroparu and SE Pit by KCB. All analyses were carried out by AGAT Laboratories located in Burnaby, British Columbia.

The leachate extraction and laboratory kinetic test analytical tests results were compared for screening purposes only to the Canadian (federal) (2014) water quality guidelines and the British Columbia (provincial) water quality guidelines (2018) for the protection of freshwater aquatic life.

Sona Hill Static Test Results

Overall, the sulphur content (total and sulphide) of the Sona Hill Deposit lithologies and rock types sampled and tested is low; predominantly below 1 wt.%. WEATH, VOLV, VACI, INTRUS, HYDR and CAYTHDR lithologies have sulphide-sulphur contents predominantly less than 0.1 wt.%, considered to be below the lower sulphide-sulphur threshold for generating ARD. The DYKE and QZVN lithologies had a sulphide-sulphur content predominantly between 0.1 wt.% and 1 wt.%. The higher sulphide-sulphur content of these lithologies indicates that sulphide oxidation could potentially release higher concentrations of secondary reaction products including acidity, sulphate, and metals. However, the paste pH values, and acid soluble sulphate-sulphur concentrations suggest that there are low stored acidity and sulphate from sulphide oxidation in these lithologies and rock types.

As determined by the modified Sobek Neutralization Potential (mS-NP) values, overall, there is moderate NP in lithologies and rock types from the Sona Hill Deposit. As expected, WEATH lithology has negligible NP due to its more intensely weathered nature. QZVN lithology also had lower levels of NP, reflecting a low carbonate content.

The NPR or mS-NP/SAP) is predominantly greater than 2.0 and therefore have an ARD classification of NPAG. However, some instances of NPR values between 1 and 2 were determined for HYDR and QZVN lithologies, indicating an Uncertain (UC) ARD classification. Similarly, some instances of NPR values below 1 were determined for HYDR, INTRUS, QZVN and WEATH lithologies, indicating a PAG ARD classification. The ARD risk classification has been adjusted for some samples based on the results of NAG testing. A QZVN and HYDR sample were reclassified from UC to NPAG and a WEATH sample reclassified from PAG to NPAG.

The of screening solid-phase elemental results to three times average crustal abundance for high-calcium granite, indicated that Au, Ag, Co, Cr, Cu, Mg, Mn, Mo, Ni, phosphorous (P), sulphur (S), selenium (Se) and tungsten (W) are elevated in one or more samples.

Sona Hill Leachate Extraction Test Results

The Shake Flask Extraction (SFE) tests generated leachate pH that was between 6.5 and 9.0 for all samples except WEATH sample 951907, which yielded a weakly acidic pH of 4.99. The sulphate concentrations of all samples and lithologies were low. With the exception of 1 sample, all were below 10 mg/L.

Arsenic, boron, chromium, cobalt, copper, iron, lead, manganese, molybdenum, nickel, silver, thallium, tungsten, zinc, and mercury were liberated at concentrations below guidelines and in many instances below the detection limit. At a neutral pH range, these elements are not expected to be readily soluble in seepage, surface water run-off or mine drainage.

Aluminum was elevated above guidelines in SFE leachate for four of ten samples by up to approximately four times. Aluminum can be expected to be readily released from HYDR, INTRUS, VACI and VOLC lithologies. However, under site-specific field conditions, the aluminum concentrations may be lower and/or controlled by secondary mineral precipitation such as gibbsite.

Cadmium exceeded the Canadian Council of Ministers of the Environment (CCME) guideline from WEATH lithology and is thus considered a Potential Constituent of Concern (PCOC) under neutral pH conditions from this lithology. However, it should be noted that WEATH lithology is capable of generating acidic pH conditions that will influence the solubility and mobility of other PCOC's not identified here.

Selenium exceed both guidelines CATHYDR lithology and is thus considered a PCOC under neutral pH conditions from this lithology.

The NAG test generated leachate pH that was above 9.0 (9.27 to 10.5) in four of seven samples that included CATHYDR, DYKE, HYDR and INTRUS lithologies, and thus above pH screening guidelines. DYKE, QZVN, and WEATH lithology samples were within the pH screening guideline range of 6.5 to 9.0. The sulphate concentrations of all samples and lithologies were low, with the exception of QZVN sample 904702, which generated 163 mg/L.

Arsenic, boron, cobalt, iron, lead, manganese, molybdenum, nickel, thallium, tungsten, zinc, and mercury were liberated at concentrations below guidelines and in many instances below the detection limit. At an alkaline pH range, these elements are not expected to be readily soluble in seepage, surface water run-off or drainage.

Aluminum was elevated above guidelines in NAG leachate for five of seven samples by up to approximately seven times. Aluminum can be expected to be readily released from CATHYDR, DYKE, HYDR, INTRUS and WEATH lithologies.

Chromium exceeded the CCME guideline for all seven sample and is thus considered a PCOC under alkaline pH conditions. Similarly, Cu concentrations in NAG liquor was up to several orders of magnitude higher than both guidelines (0.002 mg/L) and is thus considered a PCOC.

Selenium exceed both guidelines in three (CATHYDR, HYDR and INTRUS) of seven samples and is thus considered a PCOC under alkaline pH conditions from these lithologies.

Sona Hill Static Laboratory Kinetic Test Results

The results of laboratory kinetic test for the selected CHYDR/CATHYDR sample has indicated that: (1) it is not likely to become acid generating based on a lower sulphur content and moderate NP content, and (2) under neutral pH conditions, selenium may be released at rates resulting in elevated concentrations that may exceed acceptable water quality criteria.

The results of HCT for the selected QZP/QZVN sample, representative of waste rock and economic material, has indicated that: (1) it may become acid generating at some point in the future if the Neutralization Potential is depleted prior to sulphide exhaustion, and (2) under neutral pH conditions, silver, copper, and zine may be released at rates resulting in elevated concentrations that may exceed acceptable water quality criteria.

The results of HCT for the selected SAP/WEATH sample has indicated that: (1) it is not acid generating due to the negligible sulphur content, and (2) will release slightly acidic drainage, which may contain elevated Cu concentrations that may exceed acceptable water quality criteria.

Based on estimation of times to sulphide-sulphur exhaustion and carbonate depletion from the laboratory kinetic tests, the onset of ARD is not expected for CHYDR/CATHYDR, QZP/QZVN and SAP/WEATH lithologies/rock types.

20.1.4 Known Environmental Issues

Although additional studies are recommended to further develop mining waste management strategies and characterize the PEA-proposed expansion areas, there do not appear to be any known environmental issues that could materially impact the Company's ability to extract the mineral resources or reserves at the site in an environmentally responsible manner that eliminates or minimizes potential environmental risks to the receiving environment. Preliminary mitigation strategies and management plans have been developed to reduce environmental impacts to meet regulatory requirements and the specifications of the environmental permit. However, it is recommended that an updated to the KCB water quality model (2017) be undertaken in concert with Feasibility engineering design to ensure that water management structures are designed to ensure discharge to the receiving environment adequately protects the receiving environment during all phase of the mine life.

20.2 Operating and Post-Closure Requirements and Plans

The overall environmental management objective of the Project is to use BATs, BMPs and modern, proven technology to operate a gold and copper mine, process plant, and supporting infrastructure consistent with the social, economic, and environmental requirements of the Government of Guyana and, to the extent that they represent recognized international BMPs and World Bank/IFC/Equator Principles policies and guidelines.

The Company will establish and maintain a documented, comprehensive ESMS over the construction, operation, and closure phases of the Project. The ESMS will be based on current World Bank Group/International Finance Corporation guidelines.

The environmental permit requires a number of operating plans, including the Environmental Management Plan components listed in the EIA:

- Open Pit Management
- Underground Management
- Overburden Management
- Water Management
- Tailings Pond Management
- Hazardous Materials Management
- Explosives Management
- Cyanide Management
- Waste Management
- Spill Contingency Plan
- Catchment Area Management
- Social Management Plan
- Erosion and Sediment Control Plan
- Land Reclamation and the Road Management Plan

The permit requires that a Health Safety and Environmental (HSE) officer be employed and be responsible for the implementation of the Environmental Management Plan. In addition to the plans listed above, the permit contained requirements for biodiversity protection and air quality management.

The International Cyanide Management Code for the Manufacture, Transport and Use of Cyanide is specified to be applied to the use of cyanide in the mine processing. Progressive reclamation and closure as outlined in the EIA are required.

A number of monitoring and reporting requirements were established in the environmental permit, including submittal of the final design of the facilities, quarterly sampling reporting, an annual Mine Plan, recordkeeping, annual environmental reporting, emergency notification and other requirements.

20.3 Project Permitting Requirements

The Mining Act of 1989 governs the establishment of a mine and appoints the GGMC as the state agency with responsibility for mining in Guyana. In addition to the Mining Act; the Amerindian Act, the Environmental Protection Act and the Occupational Health and Safety Act also set out conditions relevant to the development of a mine.

For large scale operations, the operator is required to apply for a mining license. The process for the application of a mining license requires that the applicant submit a technical and economic feasibility study, processing and mine plans, and an EIA. A mining license is valid for 20 years, or for the life of the mine if it is shorter and can be renewed at the end of the first 20 years, if needed. A mining license is only granted after all the prerequisite conditions have been met. The license holder must pay an annual rental fee for each acre within the mining permit. The rate for a mining permit is set out by the GGMC and updated periodically. In some cases, a performance reclamation bond may be required.

20.3.1 Permitting Status

The Project is subject to a number of regulatory permits and licenses, issued by several different governmental agencies. The Project received environmental permits for gold and copper mining and processing based on an original permit application dated May 2, 2008, and the approved EIA (ETK, 2012). The permit included design, operational and reporting compliance items.

The applicable primary permit or license requirements, and the status of any permit applications, are summarized in Table 20-3.

Table 20-3: Environmental Permits and Authorizations

Name	Permitting Agency	Status	Comments
Prospecting and Mining Permits and Licenses	Guyana Geology and Mines Commission (GGMC)	Active	ETK has rights to a number of permits and license as described in Section 4 of this report.
Environmental Permit	Guyana EPA	Issued for June 2012 – May 2017 (renewed in 2017)	Environmental Permit 20050201-ETKIO was granted by the Guyana EPA after reviewing the final EIA prepared by ETK Inc (2012). The Permit was renewed in August of 2019.
Mining License	GGMC	Pending	This Technical Report to be submitted for review
Permit to use cyanide	GGMC	Pending	Before commencing any use of cyanide, the operator must apply for a special cyanide permit from GGMC, providing information on: • The site, design or process, and amount of cyanide to be used. • Site characteristics and layout. • Distance to water bodies. • Groundwater regime. • Method of tailings disposal. • Possible impacts on the environment. • General description of the activity. • Strategies for minimizing the use of cyanide over the long term.
Permit to transport, store, handle and use explosives	GGMC and Guyana Police Force	Pending	GGMC and Guyana Police Force must approve a Blasting Management Plan for the Project, as well as the design and construction of on site magazines and bulk explosive mixing systems.
Permit to operate solid waste landfills	Guyana EPA, Ministry of Health, and Central Housing and Planning Authority	Pending	

Source: Sandspring, 2019

ETK submitted an amendment to its Environmental Management Plan in October of 2021 to include the processing of silver from the deposits and adding the Southeast Area of the Main Toroparu Deposit and the Sona Hill Deposit to the permitted operations under the Environmental Authorization. EPA accepted the revised Environmental Plan on November 22, 2021.

20.3.2 Post-Performance or Reclamations Bonds

The Guyana Geology and Mines Commission and the Environmental Protection Agency are the two main government entities that are responsible for ensuring that the mining company properly executes the closure and reclamation plan in keeping with the regulations. The GGMC maintains an environmental bond which is not returned to the Company unless the mine site has been properly closed and restored.

No post-performance or reclamation bond was specified in the approved EIA issued by the Environmental Protection Agency; however, a detailed closure plan is required two years prior to scheduled closure and the plan will be subject to agency approval. A bond may be specified and required as part of the modification process for the proposed PEA operation and amended EIA.

20.4 Social and Community

The socio-economic and socio-cultural baseline was compiled based on literature review and on field surveys conducted in communities considered to be within the Project area of influence. The study details and interpretation were presented in the EIA (ETK, 2012).

The EIA presents a summary of the impacts, the rating and recommended mitigation measures, which will be further detailed in the proposed Social Management Plan. The mitigation measures reduced all major impacts to minor categories as shown in Table 20-4.

Table 20-4: Summary of Socio-cultural Impacts and Mitigation Strategies

Phase	Impact	Source of Impact and Existing Vulnerability	Impact Rating Prior to Mitigation or Prior to Implementation of ESMP	Recommended Mitigation Measures – Proposed Social Management Plan	Impact Rating after Mitigation or After Implementation of ESMP
Social Resources					
Construction	**Potential influx into the Area**	*Source* • Development of mine • Potential employment opportunities • Non streamlined recruitment process • Local vendors • Service providers/Prostitutes *Vulnerability* • Pressure on existing resources like land, water, forest use, availability of goods and services • Security influx can create security issues and increased crime • Influx of people from outside could create health risks to the workers and vice versa. Influx of sex workers often leads to rise in HIV and other STD • Social interaction with other groups is likely to bring about an increase in alcohol and drug abuse, prostitution and crime • Circulation of money from wages and salaries may increase demand for alcohol, drugs and sexual services for migrant and expatriate workers	Major (Severity – high; Likelihood – medium)	1. Discourage influx into area for employment by having a clearly established hiring policy 2. Conduct information dissemination campaign on hiring practices 3. Undertake disease prevention training. Integrate communicable diseases and STD into fact sheets in H&S training for employees **and** contractors 4. Make HIV and STD testing and counseling available to workforce, contractors on a voluntary basis 5. Conduct awareness campaigns on HIV/AIDS for the workforce	Minor (Severity – low; Likelihood – medium)
Construction	**Increase Traffic and safety risks**	*Source* • Air, Barge and overland transport of personnel, construction materials and supplies to mine site *Vulnerability* • Local communities and artisanal miners using Essequibo River • Equipment and materials being transported in barges on river • Project employees	Major (Severity – medium; Likelihood – high)	1. Enforcement of Traffic Management Plan	Minor (Severity – low; Likelihood – medium)

Source: ETK, 2012

Phase	Impact	Source of Impact and Existing Vulnerability	Impact Rating Prior to Mitigation or Prior to Implementation of ESMP	Recommended Mitigation Measures – Proposed Social Management Plan	Impact Rating after Mitigation or After Implementation of ESMP
Social Resources (continued)					
Construction	**Expectations/ concerns and management of local issues**	*Source* Unaddressed socioeconomic issues in Itaballi and communities serving as access points to project. The key issues are: • Expectations of jobs/opportunities from ETK • Expectations that project will promote local employment and community development • Benefits associated with mining and dis-benefits of mining such as deforestation or pollution which has historically been their experience • Unemployment and lack of economic opportunities for residents of Itaballi and other surrounding communities; Bartica etc. *Vulnerability* • Contentment of residents of Itaballi and surrounding communities such as Bartica etc. • Good relations with Itaballi and surrounding communities such as Bartica etc.	Moderate (Severity – medium; Likelihood – medium)	1. Dialogue with Itaballi and other surrounding communities to understand their expectations 2. Providing employment and other opportunities to residents of Itaballi and other surrounding communities	Minor (Severity – low; Likelihood – medium)
Construction	**Potential conflicts with artisanal miners**	*Source* • Potential for artisanal miners in the area as a result of a developed mine area *Vulnerability* • Security of ETK workers • Good relations with artisanal miners	Moderate (Severity – medium; Likelihood – medium)	1. Consultation with artisanal miners. 2. Active patrols of concession boundaries.	Minor (Severity – low; Likelihood – medium)
Operation	**Reduction in employment and other induced economic benefits**	*Source* • 'Boom and bust' effect. In the construction phase there is high demand for workforce as well as opportunities for local service providers. That demand will reduce in the operations phase. *Vulnerability* • Local economy and employment rates • Unskilled and semi-skilled labor • Demand for local goods and services	Major (Severity – high; Likelihood – medium)	1. The company will develop sustainable strategies and diversify the skills of its workers and service providers to enable them to find economic opportunities elsewhere in the country 2. A focused closure plan will be developed to address the issue of economic loss with appropriate mitigation measures	Minor (Severity – low; Likelihood – medium)

Source: ETK, 2012

Phase	Impact	Source of Impact and Existing Vulnerability	Impact Rating Prior to Mitigation or Prior to Implementation of ESMP	Recommended Mitigation Measures – Proposed Social Management Plan	Impact Rating after Mitigation or After Implementation of ESMP
Social Resources (continued)					
Operation	**Increased Traffic and Safety Risks**	*Source* • Upgraded 'ETK Road' to Mine Site *Vulnerability* • Trucks/other vehicles carrying equipment along the 'ETK Road' • Artisanal miners • Company employees	Moderate (Severity – medium; Likelihood – medium)	1. Partnership with the Government and other road users to enforce Traffic Management Plan	Minor (Severity – low; Likelihood – medium)

Source: ETK, 2012

20.5 Mine Closure

The license holder is responsible for mine closure and reclamation. The company must conduct closure and restoration according to the agreements signed with the GGMC. For large operations, a detailed mine closure and restoration plan is typically developed during the EIA (as was prepared for the Project in the 2012 EIA and subsequent EPA Permit 20050201-ETKIO), while the medium sized operations are generally bound by the Environmental Management Plans signed with the GGMC. NOTE: EPA Permit 20050201-ETKIO also stipulates certain design, operational and reporting items, including the requirement that a number of operating plans be incorporated into the ESMS.

In addition to the EIA and permit closure discussions, KCB updated the Project Conceptual Mine Reclamation and Closure Plan in 2017. Combined, the overall intent of the closure plan is to achieve Project objectives for restoring the site and aquatic environment to a high ecological value. The objectives of the closure plan are to:

- Prevent, reduce or mitigate the adverse environmental effects associated with the Project;
- Provide for the reclamation of all affected sites and landscapes to a stable and safe condition;
- Provide for the return of all affected ecosystems to healthy and sustainable functioning;
- Reduce the need for long term monitoring and maintenance by designing for closure and instituting progressive reclamation;
- Provide for long term monitoring and maintenance of the sites affected by the Project as required; and
- Provide for mine closure using the most current available proven technologies in a manner consistent with sustainable development.

Performance standards to measure closure success (assumed to be achieved within 20-years post-closure) are as follows:

- Physical stability (static) to a factor of safety of 1.5 for remaining facilities;
- Biological stability on 70% of site areas intended for revegetation;
- Chemical stability of mine wastes to prevent water degradation and impacts to humans or wildlife; and
- Water quality similar to or improved when compared to background pre-mining baseline data.

20.5.1 Reclamation Measures During Operations and Closure

During operations, progressive reclamation activities include:

- Progressive regrading of the Waste Rock Pile surface to reduce ponding;
- Establishment of erosion protection (e.g., riprap, revegetation) of Waste Rock Piles, TSF and other identified mine features;
- Reclamation of unused disturbance areas such as exploration roads and pads; and
- Revegetation of inactive mine features (e.g., waste rock pile benches and surfaces).

Reclamation measures at closure for each key component are summarized in Table 20-5.

Table 20-5: Facility-Specific Conceptual Closure Activities

Component	Closure Activities
Tailings Storage Facility	During operation, the embankment crest will be raised above the PMF peak elevation with 3 m freeboard; Reclaim barge/pumps, as well as tailings and water reclaim lines will be removed. The main tailings line will be flushed with freshwater, then cut into manageable sections, and dispose of in inert waste cell in final waste rock stockpile or in a separate disposal site within the TSF; Spillways for ultimate conditions will be constructed to meet the requirements of the closure spillways, which will route and discharge excess water as per the TSF design plan; Saprolite will be placed at a depth of 0.3 m as a growth medium cover over tailings. Saprolite will be sourced from the waste material from the open pit or stockpiles from TSF construction. The Project operating cost will cover part of the mining and transportation of this material. The closure plan assumes the remaining transportation and spreading cost. Erosion of the upstream and downstream exposed surface slopes of the dams will be controlled with a layer of crushed rock (rip rap) on a heavy duty geotextile. This protection layer will be placed on the upstream and downstream slopes of the starter dams and progressively as the dams are raised. This will be part of progressive and final closure works of the TSF. It is anticipated that during operations, all slope protection, sloping and additional erosion protection measures identified in the Project Erosion and Sediment Control Plan are implemented. Downgradient surface water and groundwater monitoring will continue to ensure continuing compliance with effluent guidelines.
Waste Rock Piles	To promote drainage during operation, competent rock will be placed preferentially upslope of valleys and low spots of the underlying footprint of the Waste Rock Piles. A series of catchment and diversion ditches will be constructed to collect surface run-off from the Waste Rock Pile areas. The ditches will be designed to control sediment loading into the natural water drainage, to minimize erosion of surface materials, and to divert any metal-rich waters as required. During operations, saprolite will preferentially be encapsulated in the inner parts of the Waste Rock Piles to enhance stability and avoid ARD generation a saprolite layer (0.3 m thickness) will be applied to the top lift and to selected areas of the Waste Rock Pile faces above the PMP flood level.
Open Pit	Remove loaders, drill rigs, and surplus haul trucks and mobile equipment within the pit; Disconnect and remove power and water lines from the site; install proximity warning signs to communicate potential safety hazards to the public; Construct safety berm around pit to prevent vehicular access; Establish spillway to enable discharge to surface water features around the pits. Discharge spillway will be sloped and will be covered with geotextile and rip rap (waste rock) to minimize the possibility of erosion; Breach operational water diversion berms and levees to promote safe pit flooding; and Monitor pit lake water quality and filling in order to confirm conditions will support compliance with mine effluent guidelines when the pit lake is at its design height and the outflow channel is operational.
Process Plant and Ancillary Facilities	At least two years prior to plant site and workshop demolition, assess volumes of hydrocarbons and hazardous substances (e.g., cyanide and other reagents), to reduce the volume left on site which will require removal; Process final economic material stockpile within the mill; Mobile structures and equipment will be removed at closure and sold, or deposed in an appropriate waste disposal or landfill facility; Survey for hazardous materials and wastes and remove to safe/secure storage pending final disposal as required. Return unused stocks of cyanide and other reagents to the supplier for safe disposal or credit; For cyanide related storage and piping infrastructure, conduct a third party assessment and management plan for decommissioning of all cyanide facilities in accordance with Standard of Practice 5.2 of the International Cyanide Management Code (ICMC). This may include treating flush cyanide storage tank and cyanidation plant with freshwater until WAD cyanide values <0.5 mg/l at inlet to detoxification plant, draining cyanide storage tank and plant piping, draining cyanide storage tanks and plant piping and flush through the detoxification plant to the TSF; Conduct assessment for any remaining hazardous materials and safely remove off site; Prior to demolition, remove equipment and infrastructure which has a potential saleable credit. Structures will then be dismantled or demolished and removed from site. Concrete footings and slabs may be left in place but ripped prior to placement of saprolite. Basements, sumps and/or man-made holes will be backfilled with inert rock or fill from site. Building sites will be covered with saprolite which will promote growth of secondary native vegetation; sites will be graded to blend in with existing topography, and compacted areas Will be ripped, covered with saprolite which will promote secondary native vegetation; Saprolite of 0.3 m thickness will be placed as growth medium cover; and

Component	Closure Activities
	Locally available vegetation will be used to revegetate disturbed areas.
Roads and General Land Disturbance	Mine roads, including haul roads, will be decommissioned once site access is no longer required. Decommissioning of roads will be achieved by removing culverts that do not form part of the permanent water management system. Original watercourses will be re-established to the extent possible while maintaining geotechnical and hydraulic stability Road culverts will be replaced with competent NPAG rock cross-ditches/swales and road surfaces will be ripped to improve soil structure. Internal roads will be recontoured to facilitate vegetation growth and re-establish drainage. All exploration roads at the mine site will be reclaimed in a similar manner to haul and access roads. Removal of all explosive materials will be carried out by authorized personnel and returned to the supplier Demolish explosives magazine housing. Concrete footings and slabs of explosives magazine may be left in place but ripped.
Site Water Management Structures	Open pit dewatering stops (end of mine operations); Decommissioning the sedimentation ponds and berms (end of year 1 after open pit mining); Breach the levees at specific locations (end of year 1 after open pit mining); Install erosion protection for inflow into /outflow from the pits (end of year 1 after open pit mining); Breach the Wynamu diversion dam (end of year 2 following open pit mining stopping); and Breach the Wynamu dam up to an elevation such as water is no more diverted to the diversion ditch (i.e., a smaller pond upstream is still formed). The Wynamu course is thus re-established.

Source: KCB, 2017

20.5.2 Closure Monitoring

Post-closure monitoring and maintenance will be conducted to assess performance against closure objectives. If the site is safe, stable, and non-polluting, in accordance with the identified success criteria, reclamation outcomes are assessed as successful.

Closed facilities will be inspected, and annual reports provided to evaluate the success of progressive reclamation. Reclamation monitoring would be coordinated with the EPA and Guyana Forestry Commission (GFC). The existing monitoring programs for surface and groundwater will continue in accordance with the proposed monitoring plans. Monitoring and maintenance type and frequency will be adapted to address progressive reclamation as it proceeds.

Post-closure discharge water quality is predicted to have minor exceedances of only chromium and arsenic IFC effluent guidelines under the Base Case Model results. No water treatment plant is envisaged at closure. Environmental monitoring is assumed to continue for at least 20 years, or until non-hazardous conditions are achieved for any discharge from the remaining facilities and the groundwater and surface water quality meets applicable regulatory standards. Monitoring records will be maintained by the mine operator.

The GGMC will be advised of the results of the reclamation monitoring. Stakeholders in communities in proximity to operational areas will be mobilized to support the monitoring program, where skills and experience permit.

20.5.3 Reclamation and Closure Cost Estimate

The PEA anticipates cessation of milling and processing in Year 24, with a closure cost expenditure occurring entirely in Year 25. The base allowance for closure costs presented in the Technical Economic Model is US$22,216,000 with an allowance for a 30% contingency making the total closure cost estimate for the Project to be US$28,881,000. Nordmin did not prepare this estimate, nor were the calculations provided for Nordmin's review. However, the estimate is consistent with the reclamation cost estimate attached to the most recent closure plan (KCB, 2017), and is in keeping with other gold mining operations of similar size.

No post-performance or reclamation bond was specified in the approved EIA issued by the Guyana EPA; however, a detailed closure plan is required two years prior to scheduled closure and the plan will be subject to agency approval. A bond may be specified and required as part of the modification process for the proposed PEA operation and amended EIA.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimates

21.1.1 Summary

The total estimated initial cost to design and construct the Project identified in this report is US$355 million. Approximately US$41 million of this estimate is related to pre-stripping costs and the remainder of US$314 million is directly related to the installation of the Project site facilities and purchasing of equipment.

Initial capital will support the installation of a leaching circuit that will produce doré bars bearing gold and silver and will operate at a feed rate of 7,000 tpd, this circuit will support the operation for the first five years.

In years four and five expansion capital will be used to install a flotation circuit that will operate at a feed rate of 7,000 tpd, bringing the total project feed rate to 14,000 tpd, and will produce a copper concentrate bearing copper, gold and silver. This circuit will begin operating in year 6 and its cost is estimated at US$281 million (including expansion of the mine fleet, processing circuit, infrastructure, power and associated indirect and owner's costs). The free cash flow from the Project is estimated to self-finance this expansion.

This PEA's capital cost estimates consider the PMPA with Wheaton Precious Metals Corp. (Wheaton) for the purchase of 10% of the gold produced over the LoM at US$400 per payable ounce; and 50% of the silver produced over the LoM at US$3.90 per payable ounce. The acquisition cost of this precious metal production stream of US$138 million and is entirely included as a payment towards the initial capital. This acquisition cost is used to reduce the Project's capital requirements in the economic model and are identified in this report as PMPA Installments.

Sustaining capital is estimated at US$662 million for the LoM and will support equipment maintenance and replacement, incremental capacity increases, water management structures and tailings storage facility expansions, infrastructure maintenance and associated indirect and owner's costs.

The aggregate capital estimate is considered to be within a ±40% weighted average accuracy of actual costs. Base pricing will be in Q4 2021 US dollars, with no allowances for inflation or escalation beyond that time.

The contingency cost is based on the following factors of specific direct cost areas:

- Leaching Process Circuit: 28.60%
- Flotation Process Circuit: 26.23%
- Off site Infrastructure: 5.00%
- On site Infrastructure: 10.00%
- Water Management and Treatment: 15.00%
- Tailings Storage Facility: 15.00%
- Buildings and Ancillary Equipment: 20.00%
- Closure: 30.00%

The total contingency represents roughly 17% of the direct cost estimates from the initial and expansion capital. The contingency is included to account for unanticipated costs within the scope of the estimate. The percentage allowances were individually assessed based on the accuracy of the quantity measurement, type and scope of work, and price information for the capital cost estimate.

The estimate is based on first principles estimates based on vendor quotations and cost databases from similar projects. It does not reflect discounts for negotiated prices, bulk purchasing, or used equipment purchases where appropriate, any of which could lead to reductions in actual capital costs relative to the prices used in the capital estimate.

A summary overview of the estimate by area is presented in Table 21-1.

Table 21-1: Summary of Capital Costs by Area

PEA Capital Cost Estimates (US$M)	Scope	Initial Capital (Pre-Prod) (US$M)	Expansion (US$M)	Sustaining Capital (US$M)	LoM Capital (US$M)
Mine (Open Pit & Underground)	SRK/Nordmin	24	69	601	695
Process Plant	Metifex	95	103	-	198
Water and Tailings Management	KCB	17	-	29	45
Infrastructure	GCM Mining	64	-	6	69
Power Supply	GCM Mining	3	-	-	3
Owner's	GCM Mining	23	21	22	66
Indirect Costs	GCM Mining	52	61	-	113
Risk and Contingency	GCM Mining	36	27	4	67
Subtotal Capital Expenditures		**314**	**281**	**662**	**1,258**
Capitalized Rock Pre-Stripping	SRK	41	-	-	41
PMPA Installments	GCM Mining	(138)	-	-	(138)
Net Financing Required		217	281*	662*	1,161

Source: SRK/Nordmin/Metifex/KCB/GCM Mining, 2021. * Free Cash Flow is sufficient to finance

21.1.2 Basis for Capital Cost Estimate

Mining

SRK prepared a first principles cost model to estimate the open pit mining capital and Nordmin prepared a first principles cost model to estimate the underground mining capital. New mining equipment units to be placed in service for pre-production mining were considered as purchased in the year earlier to then be on site ready for pre-production.

Mining Equipment Initial Capital Cost Estimate Basis

The mining equipment capital cost estimate was based on the following:

- All mining units are based on new equipment purchases;
- Freight cost for mining equipment was generally estimated at 7%;
- No import duties were deemed to be applicable;
- Allowances were made for on site equipment erection costs for some units;
- Mining equipment initial capital included spare parts for major equipment units;
- Mining equipment initial capital included (non-fixture) shop tools;
- Mining equipment initial capital included a fleet dispatch system and mining department (geology and engineering) equipment; and

- Sustaining capital costs are costs incurred in purchasing new mining equipment, both additional and replacement units required, and performing mining equipment rebuilds over the LoM. Mining fleet expansion was indicated in Section 16.4.18.
- No contingency was included in the mining initial capital cost estimate (as discounts will be able to be negotiated from manufacturers' base costs used for this estimate).

The mining equipment sustaining capital cost estimate was based on the following:

- All mining units are based on new equipment purchases;
- Freight cost for mining equipment was generally estimated at 7%;
- No import duties were deemed to be applicable;
- Allowances were made for on site equipment erection costs for some units;
- Mining equipment rebuilds (overhauls) were included in mining sustaining capital costs. These were estimated based on a total of 75% of the original cost of the equipment unit over the operating life of the machine, and scheduled as three overhauls during the operating life; and
- No contingency was included in the mining sustaining capital cost estimate.

The acquisition of open pit mining equipment in years -2 and -1 was modelled as a lease in the technical economic model, the terms of the lease were assumed as the following:

- Initial Lease Deposit: 20% of Acquisition Cost
- Financed Portion: 80% of Acquisition Cost
- Annual Interest: 7%
- First Period of Payment: Year 1

<u>Processing Plant Capital Cost</u>

The process plant is designed to recover Au and Cu from mineralized material supplied by the Toroparu, Southeast and Sona Hill deposits. The flow sheet encompasses primary and secondary crushing, grinding, gravity separation CIL, and flotation to produce Cu concentrate and doré. The capital cost estimate for the initial plant and the year 6 expansion is presented in Table 21-2. The Capex estimate was prepared by Metifex and assumes the operation of a single 7,000 tpd (CIL) process circuit from years 1 to 5 and two parallel 7,000 process circuits (Flotation and CIL), totalling 14,000 tpd capacity, from year 6 to the end of the mine life.

Table 21-2: Process Plant Capital Cost

Area	Yr -2 (US$M)	Yr -1 (US$M)	Yr 4 (US$M)	Yr 5 (US$M)
Excavation & Backfill for Concrete Foundations	0.62	-	0.45	-
Concrete	12.23	-	8.88	-
Structural Steel	5.22	5.22	7.39	7.39
Mechanical Equipment & Platework	12.50	18.75	16.16	24.24
Mechanical Installation	-	18.83	-	15.10
Pipework	-	11.80	-	12.82
Electrical & Instrumentation	-	10.06	-	10.66
Buildings	0.50	-	-	-
Total	**31.08**	**64.66**	**32.88**	**70.22**

Source: Metifex, 2021

No sustaining capital costs are required for these facilities, all of the required maintenance were included in the operating cost estimate.

Tailings Storage Facility

The estimated capital costs for the starter dam construction (at Module 1) are based on the construction activities defined by KCB, as included in Table 21-3.

Table 21-3: Tailings Storage Facility Capital Cost

Area	Yr -2 (US$M)	Yr -1 (US$M)	Yr 2 (US$M)	Yr 9 (US$M)
Water and Tailings Management	8.90	7.70	13.73	14.98

Source: KCB, 2021

The basis for the sustaining capital costs is related to the construction activities required to raise the Starter Dam of Module 1 and build Module 3, and their progressive closure, as defined by KCB and as shown in Table 21-4.

Table 21-4: Tailings Storage Facility Basis for Capital Costs

Module	Cost Item
Module 1	• Accesses/diversions. • Site preparation of tailings impoundment. • Tailings starter dams to crest elevation: 133 m (Stage 1). • Spillway for Stage 1. • Supply and installation of tailings and water lines and reclaim barge. • Collection Pond 1 and spillway. • Accesses/diversions where required. • Site preparation of tailings impoundment where required. • Raise of tailings dams to ultimate crest elevation: 140 m (Stage 2). • Spillway for Stage 2. • Supply and relocation of tailings lines to raised dams (year 2).
Module 3	• Accesses /diversions. • Site preparation of tailings impoundment. • Tailings starter dams to crest elevation: 127 m (Stage 1). • Spillway for Stage 1. • Relocation of tailings and water lines. • Raise of tailings dams to crest elevation: 132 m (Stage 2). • Spillway for Stage 2. • Relocation of tailings lines. • Raise of tailings dams to ultimate crest elevation: 138 m (Stage 3). • Spillway for Stage 3. • Relocation of tailings lines.
Closure	• Final Closure of Cell 3. Placement of closure cover on exposed tailings beach and revegetation.

Source: KCB, 2021

Site Water Management

The capital cost estimate for the site water management structures is based on the construction sequence for defined by KCB as follows:

- Construction scheduled to start 2 years before start of operations;
- The Wynamu Diversion Dam (including saddle dams) and Channel will be constructed first;
- Non-Contact water ditches (NCD) will be constructed concurrent with Wynamu and Saddle Dams, to limit water contamination during construction works of proposed structures;
- Levees and Berms should be constructed in parallel with NCD to limit risk of flooding of the (under) construction ditches. The excavation material from the ditches should be used for the berms and levees, which will require careful staging and handling of saprolite;
- Contact Water ditches (CD) will be constructed after NCD works are finished. Similarly, the excavation from the CD will also be used in the Levee and Berm construction;
- Levees are expected to be constructed in two stages:
  - Startup – the crest elevation to contain the 1 in 100-year 24-hr design flood elevation; and
  - Ultimate – the crest elevation to contain the 1 in 1000-year 24 hr design flood elevation.

The sustaining capital for water management and treatment was assumed as 2.5% of the initial capital spent on a yearly basis.

Infrastructure

The capital estimate for the Project infrastructure (Table 21-5) is based on construction estimates by SRK and the Company as follows:

- Construction starts 3 years before start of operations;
- Infrastructure cost estimates are based on adjustments and updates to previous detailed engineering estimates, plus factoring, and unit cost buildups;
- Third party power cost estimates for fuel oil generation (quotations and allowances); and
- Third party hydroelectrical construction ownership costs are included as operating costs and offset fuel oil costs.

Table 21-5: Infrastructure Capital Cost

Area	Yr -3 (US$M)	Yr -2 (US$M)	Yr -1 (US$M)	Yr 1 (US$M)	Yr 2 (US$M)
Infrastructure	3.47	31.69	28.56	1.23	4.46

Source: GCM Mining, 2021

Owner's and Indirect Costs

Owner's and indirect costs estimates were prepared (Table 21-6) under the following assumptions:

- Owner's
  - Pre-Commissioning: 0.6% of process plant, on site infrastructure, water management and treatment and tailings management;
  - Commissioning: 0.4% of process plant, water management and treatment and tailings management;
  - Spares: US$2.8 million;
  - First Fills: 1.6% of direct capital costs;
  - Owner's: 6.6% of direct capital costs; and

- EPCM: 4% of process plant, on site infrastructure, water management and treatment and tailings management.
- Indirect costs were estimate as part of Metifex's cost model

No sustaining capital costs are required for this item.

Table 21-6: Owner's and Indirect Capital Cost

Area	Yr -2 (US$M)	Yr -1 (US$M)	Yr 4 (US$M)	Yr 5 (US$M)
Owner's	6.24	16.59	10.68	10.68
Indirect Costs	26.07	26.07	30.35	30.35

Source: GCM Mining, 2021

21.2 Operating Cost Estimates

21.2.1 Summary

The PEA estimate is based on first principle calculations supported by cost databases on similar operations. Operating costs have been prepared in Q4 2021 US dollars and exclude:

- Contingency;
- Escalation;
- Taxes (VAT); and
- Import Duties.

Imported equipment, materials, and operating supplies are not subject to VAT, import or other duties as per the Mineral Agreement with the Government of Guyana.

The operating cost estimates have been assembled by area and component, based upon estimated staffing levels, consumables, and expenditures according to the mine and process design. LoM operating costs are shown in Table 21-7, and annual operating costs in Table 21-8 (rounded to nearest US$1,000,000).

Table 21-7: Operating Cost LoM

Area	Expenses (US$M)	US$/t Mined	US$/t-Mill
Mine	1,841	2.65	17.16
Processing	1,558	n/a	14.52
G&A	360	n/a	3.36
Total Operating	**3,758**	**n/a**	**35.03**

Source: SRK, 2021

Table 21-8: Annual Operating Cost, US$ x 1,000,000

Period	Ore Milled (Mt)	Mining (US$M)	Processing (US$M)	G&A (US$M)	Total (US$M)	US$/t milled
-3	-	-	-	-	-	-
-2	-	-	-	-	0	-
-1	-	-	-	-	0	-
1	2.18	(42)	(35)	(14)	(91)	(41.99)
2	2.56	(43)	(39)	(14)	(96)	(37.45)
3	2.56	(42)	(38)	(15)	(95)	(37.19)
4	2.56	(45)	(39)	(15)	(99)	(38.67)
5	2.56	(44)	(40)	(15)	(99)	(38.79)
6	4.73	(41)	(69)	(16)	(126)	(26.71)
7	5.11	(45)	(74)	(17)	(135)	(26.50)
8	5.11	(54)	(74)	(17)	(144)	(28.17)
9	5.12	(55)	(74)	(17)	(145)	(28.28)
10	5.11	(56)	(73)	(17)	(146)	(28.63)
11	5.11	(96)	(73)	(17)	(186)	(36.36)
12	5.11	(116)	(73)	(16)	(205)	(40.18)
13	5.12	(114)	(74)	(16)	(204)	(39.75)
14	5.11	(120)	(73)	(16)	(209)	(40.97)
15	5.11	(119)	(73)	(14)	(207)	(40.46)
16	5.11	(115)	(73)	(14)	(202)	(39.63)
17	5.12	(117)	(73)	(14)	(205)	(40.00)
18	5.11	(113)	(73)	(14)	(201)	(39.28)
19	5.11	(112)	(73)	(14)	(199)	(39.04)
20	5.11	(108)	(73)	(14)	(195)	(38.19)
21	5.12	(110)	(73)	(14)	(197)	(38.44)
22	5.11	(85)	(74)	(13)	(173)	(33.86)
23	5.11	(39)	(74)	(13)	(125)	(24.56)
24	3.25	(9)	(50)	(14)	(72)	(22.25)
25	-	-	-	-	0	-
Total	**107.30**	**(1,841)**	**(1,558)**	**(360)**	**(3,758)**	**(35.03)**

Source: SRK, 2021

21.2.2 Basis for Operating Cost Estimate

Mining

Mine operating costs were developed by SRK and Nordmin and based on the mine plan, equipment requirements, and manpower requirements, parts of which have been presented in previous sections. The basis of the operating costs is an owner operated mine. The mine operating costs include all the supplies, parts, and labour costs associated with mine supervision, operation, and equipment maintenance.

SRK estimated the required mining equipment fleets, required production operating hours, and manpower to arrive at an estimate of the mining costs. The open pit mining costs were developed from first principles. The mining operating costs are presented in the following categories:

- Production Drilling;
- Production Blasting;

- Production Loading; and
- Production Hauling.

A maintenance cost was allocated to each category that requires equipment maintenance. The operating costs are defined as starting in Year 1 and exclude any pre-production operations.

Nordmin estimated the required mining equipment fleets, required production operating hours, and manpower to arrive at an estimate of the mining costs. The underground mining costs were developed from first principles. The mining operating costs are presented in the following categories:

- Labour;
- Maintenance Parts;
- Consumables;
- Cement;
- Diesel;
- Infill Drilling (Contractor);
- Surface Support; and
- Miscellaneous.

The mining costs may be referenced as per tonne mined (waste and economic material tonnes mined basis), and as per economic material tonne mined, (note the latter is not necessarily the same as per economic material tonne processed in the same year due to stockpile economic material re-handling). By “per tonne mined” is meant as excavated from the open pits and does not include rehandled stockpile economic material.

Employee classifications, wages and burden benefits are based on information provided by the Company. The costs for maintenance supplies and materials were based on estimates presented in the current Infomine mining cost service publications.

It was assumed that the Toroparu Mine will not incur duties on imported equipment and supplies.

A summary of the mining operating costs, per tonne mined for each year is presented in Table 21-9 and Table 21-10.

Table 21-9: Open Pit Mining Operating Cost, US$/t moved

Year	Drilling (US$/t)	Blasting (US$/t)	Loading (US$/t)	Hauling (US$/t)	Roads & Dumps (US$/t)	Total (US$/t)
-1	0.22	0.48	0.38	0.50	0.25	1.82
1	0.23	0.48	0.39	0.59	0.25	1.93
2	0.23	0.48	0.40	0.56	0.25	1.92
3	0.23	0.48	0.40	0.64	0.28	2.02
4	0.23	0.48	0.40	0.67	0.27	2.04
5	0.24	0.47	0.41	0.66	0.27	2.06
6	0.22	0.49	0.24	0.56	0.23	1.74
7	0.22	0.49	0.26	0.65	0.23	1.85
8	0.22	0.49	0.28	0.69	0.22	1.88
9	0.23	0.48	0.28	0.75	0.22	1.97
10	0.24	0.48	0.28	0.85	0.23	2.07
11	0.19	0.49	0.28	0.79	0.21	1.97
12	0.21	0.48	0.28	0.82	0.19	1.97
13	0.23	0.47	0.29	0.78	0.21	1.97
14	0.21	0.49	0.28	0.94	0.20	2.11
15	0.22	0.48	0.29	0.97	0.20	2.16
16	0.22	0.48	0.29	0.98	0.20	2.18
17	0.22	0.49	0.29	0.97	0.20	2.16
18	0.23	0.47	0.24	1.02	0.20	2.17
19	0.23	0.47	0.26	1.03	0.22	2.21
20	0.24	0.47	0.27	0.97	0.22	2.17
21	0.23	0.48	0.28	0.98	0.20	2.17
22	0.23	0.51	0.27	0.99	0.21	2.21
23	0.12	0.55	0.26	0.78	0.18	1.90
24	0.05	0.95	0.25	0.43	0.45	2.12
LoM/t moved	**0.22**	**0.49**	**0.30**	**0.82**	**0.22**	**2.05**

Source: SRK, 2021

Table 21-10: Underground Mining Operating Cost, US$/t mined

Year	Labour	Maintenance Parts	Consumables	Cement	Diesel	Infill Drilling	Surface support	Misc.	Total
11	6.34	5.34	7.87	3.08	5.28	6.11	3.75	1.20	38.96
12	6.35	6.09	11.95	5.62	6.44	6.28	3.75	1.20	47.67
13	7.09	6.24	9.78	6.28	6.81	6.28	3.75	1.24	47.48
14	6.58	5.65	12.66	5.61	6.29	5.74	2.50	1.04	46.09
15	6.58	5.65	11.12	5.85	6.29	5.74	2.50	1.02	44.76
16	6.49	5.65	10.42	6.03	6.29	5.59	2.50	1.01	43.98
17	6.35	5.65	12.33	6.08	6.29	5.52	1.88	1.07	45.17
18	6.38	5.65	11.09	6.45	6.29	5.52	1.88	1.05	44.30
19	6.41	5.50	9.63	6.70	5.92	5.52	1.88	1.02	42.56
20	6.35	5.50	9.81	6.20	5.92	5.52	1.25	1.01	41.55
21	5.76	5.03	9.56	6.21	5.86	4.76	1.25	1.01	39.45
22	4.75	3.38	2.13	3.27	3.96	3.77	-	0.53	21.80
LoM/t mined	**5.32**	**4.61**	**8.34**	**4.75**	**5.05**	**4.68**	**1.89**	**0.87**	**35.52**

Source: Nordmin, 2021

Mineral Processing and Metallurgical

The processing operating cost includes the cost of all material, consumables and labour required to process the feed from the mine. This includes all electrical power requirements, reagents, operating and maintenance supplies and labour. A summary of the process plant operating costs, per tonne milled for each year is presented in Table 21-11.

Table 21-11: Process Plant Operating Cost

Year	Labour	Power	Consumables	Maintenance	Miscellaneous	Total $/t milled
1	1.93	5.92	4.81	2.29	0.91	15.86
2	1.64	5.89	4.77	2.00	0.80	15.09
3	1.48	5.89	4.77	2.00	0.80	14.93
4	1.48	6.16	4.79	2.03	0.80	15.25
5	1.48	6.36	4.80	2.06	0.80	15.51
6	0.87	6.84	3.82	2.62	0.54	14.68
7	0.80	6.88	3.74	2.51	0.51	14.43
8	0.80	6.88	3.73	2.51	0.51	14.42
9	0.72	6.88	3.73	2.51	0.51	14.35
10	0.72	6.88	3.74	2.51	0.51	14.35
11	0.72	6.88	3.74	2.51	0.51	14.35
12	0.72	6.88	3.73	2.51	0.51	14.34
13	0.72	6.88	3.74	2.51	0.51	14.35
14	0.72	6.88	3.72	2.51	0.51	14.33
15	0.72	6.88	3.70	2.51	0.51	14.32
16	0.72	6.88	3.72	2.51	0.51	14.33
17	0.72	6.88	3.72	2.51	0.51	14.34
18	0.72	6.91	3.72	2.51	0.49	14.34
19	0.72	6.88	3.73	2.51	0.49	14.32
20	0.71	6.88	3.72	2.51	0.49	14.31
21	0.64	6.95	3.73	2.52	0.49	14.33
22	0.64	7.11	3.75	2.54	0.49	14.53
23	0.57	7.12	3.72	2.54	0.49	14.44
24	0.62	6.62	3.92	3.49	0.66	15.31

Source: Metifex, 2021

General and Administrative

The G&A for the Project was estimated for each year of operation. The cost varies between US$9 and US$16.7 million per year or US$3.36/t processed on average over LoM. The basis for the G&A personnel structure is the experience gained by the Company’s management from other operating mines.

Labour salaries of the admin and logistics area were incorporated in the G&A estimate, these salaries vary from US$19,344 to US$251,344.

Camp expenses have been calculated on a unit cost per-man, per day basis. The first step in producing this figure is to calculate the number of individuals on site. This has been done by taking into consideration the number of employees required at each position and their assigned rotation schedules outlined in the organizational chart. The number of people is then assigned to one of three cost centres: G&A, mine and process.

A number of fixed expenses are also considered when calculating G&A costs and the rest of the G&A cost is composed of miscellaneous items that include non-personnel administration related costs, such as:

- Miscellaneous indirect costs such as safety supplies, medical supplies, life insurance, general training, recreation and office supplies;
- Light vehicles, operation and maintenance (for administration only);
- Travel, meetings, conferences and training and recreation programs (HR activity);
- Personnel transportation;
- First aid centre related costs (operating cost beyond labour cost);
- Environmental (permitting, monitoring, hydrology, equipment, reclamation and various other commitments);
- Insurance (property and business interruptions, buildings and equipment, liability);
- Communication and other public relations activities;
- Georgetown office expenses;
- Accommodations;
- Various legal fees;
- Software/hardware (for administration only);
- Warehouse expenses; and
- Miscellaneous consulting.

These fixed costs are added to the total G&A costs for each year.

22 ECONOMIC ANALYSIS

22.1 Method of Evaluation

A yearly discounted cash flow model was created to evaluate the Project assuming the Project is 100% equity financed. The analysis assumes the terms of the precious metal purchase agreement with Wheaton.

Mine production assumptions were developed by SRK and Nordmin and plant production assumptions were developed by Metifex, Water Management Structures and Tailings Storage Facility assumptions by KCB, with Infrastructure, Power Generations, Owners Costs, Sustaining Capex and Contingencies provided by the Company.

Mining cost estimates were provided by SRK and Nordmin and process costs were provided by Metifex and reviewed by SRK. Off site infrastructure costs and Owner's costs were provided by the Company. Additional costs such as refining, royalties and administrative costs provided by the Company were subtracted from the revenue to calculate an estimated cash operating margin.

SRK and the Company prepared a detailed financial model presented in Appendix E estimating cash flows by year for the forecast mine life.

All revenues and costs are expressed in Q4 2021 US dollars.

22.2 Input Parameters

The proposed Project including the open pits, underground mine, processing facility and on site and off site infrastructure would be developed by the Company with assistance from EPCM contractors and suppliers. The contractors would assist the Company in port development and the construction of the camp, processing, HFO power generation facility, TSF and other infrastructure. The open pits would be developed by the Company using its own labour force and equipment. The open pits, underground mine, processing facility and on site and off site infrastructure, logistics including concentrate transportation to the port, port operation and barge loading would be operated and maintained by the Company using its own labour force and equipment with the assistance of equipment maintenance specialists; geotechnical consultants, an explosive supplier; and other specialists. The key criteria, principal assumptions and input parameters used in the Base Case are shown in Table 22-1.

The major input parameters to the model include Au, Ag and Cu prices, initial and sustaining capital, operating costs, mining rates, and estimated taxes and royalties. Additionally, several minor assumptions throughout the model such as working capital, environmental accruals and depreciation rates affect the estimated project economics to a lesser degree.

SRK and Nordmin prepared the mine production schedule that will support the operations. The mineable part of the resources was categorized into four different mining types, including:

- Saprolite: Softer material type that can be fed to the leaching circuit;
- Fresh Rock Leaching: Fresh rock material type that should only be fed to the leaching circuit;
- Fresh Rock Flotation: Fresh rock material type that should only be fed to the flotation circuit; and
- Fresh Rock Flex: Fresh rock material that can be fed to either the leaching or the flotation circuit. This material is used to adjust the feeding schedule and use the full capacity of each circuit.

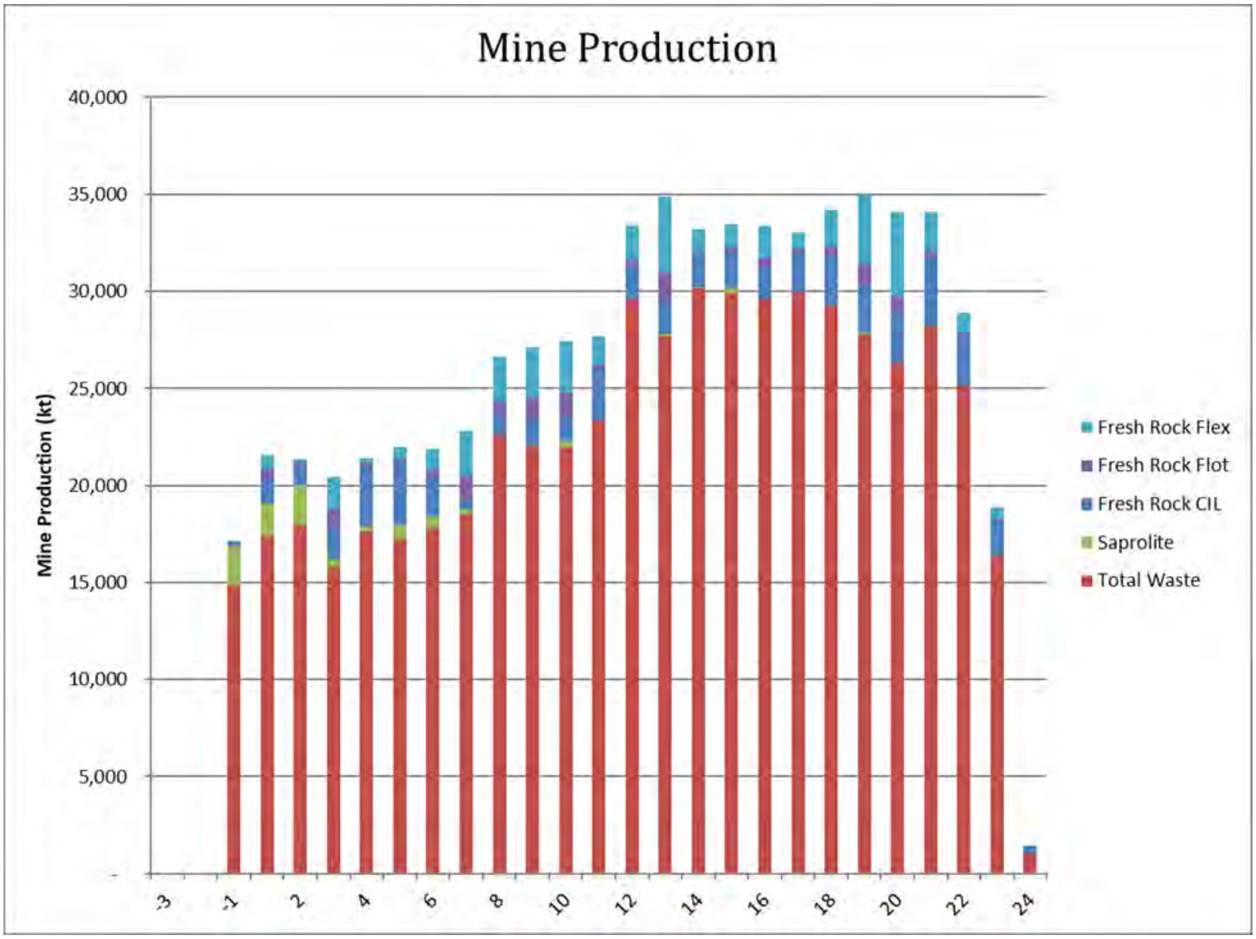
Table 22-1 presents the summary of the Project's mineral resources within the PEA mine plan.

Table 22-1: Toroparu Mineral Resources within the PEA Mine Plan Summary

Description	Value	Units
Waste Mined	**557,943**	**kt**
Mined Resource	**107,302**	**kt**
Saprolite	9,132	kt
Fresh Rock Leaching	45,970	kt
Fresh Rock Flotation	12,361	kt
Fresh Rock Flex	39,839	kt
Mined Resource Average Au Grades		
Saprolite	1.52	g/t
Fresh Rock Leaching	1.94	g/t
Fresh Rock Flotation	0.85	g/t
Fresh Rock Flex	1.34	g/t
Mined Resource Average Cu Grades		
Saprolite	0.09%	%
Fresh Rock Leaching	0.02%	%
Fresh Rock Flotation	0.24%	%
Fresh Rock Flex	0.13%	%
Mined Resource Average Ag Grades		
Saprolite	1.22	g/t
Fresh Rock Leaching	0.55	g/t
Fresh Rock Flotation	2.07	g/t
Fresh Rock Flex	1.41	g/t

Source: SRK, 2021

The mining operations start in year -1 at a capacity close to 17.5 Mtpa and this is ramped-up to 22 Mtpa in year 1. This production rate is kept at this level until year 8, when the mining capacity is increased to 27.5 Mtpa. The mining capacity is ramped-up again in year 12 when it is increased to a maximum capacity of 35 Mtpa after the underground operation is installed and ramped-up. Figure 22-1 presents a summary of the mine production.



Source: SRK, 2021

Figure 22-1: Mine production summary

The Project's mine production is supported by three open pits, including the Main Pit, the SE Pit, the Sona Hill Pit and an underground mining operation.

The model was also based on the following Project basic schedule:

- Mining License and Financing approvals: 1 year;
- Construction period: 2 years; and
- Production period: 24 years.

The financial model assumes a two-stage construction approach: First stage is the construction of the grinding and leaching plant for the saprolite and fresh rock during Years -2 and -1. Commencement of leaching is assumed on January 1 Year 1. Second stage is the construction of the grinding and flotation plant for the fresh rock in years 4 and 5. Commencement of flotation is assumed on January 1 Year 6.

The following are brief descriptions of the designed plant phases:

- Stage 1: All of the existing milling capacity is used to process Au bearing saprolite and fresh rock; and
- Stage 2: Expand milling capacity to support flotation of Cu Au bearing fresh rock and continue leaching.

The processing rates for the stages described above are presented in Table 22-2.

Table 22-2: Project Stages

Description	Value	Units
Phase 1 Processing Rates		
Saprolite + Fresh Leach Daily Capacity	7,000	tpd
Fresh Rock Flotation Daily Capacity	0	tpd
Phase 2 Processing Rates		
Saprolite + Fresh Leach Daily Capacity	7,000	tpd
Fresh Rock Flotation Daily Capacity	7,000	tpd

Source: SRK, 2021

Plant feed type were separated by area and type and individual recoveries were applied to each of these material types. Table 22-3 presents the composition of the materials fed to the processing plants.

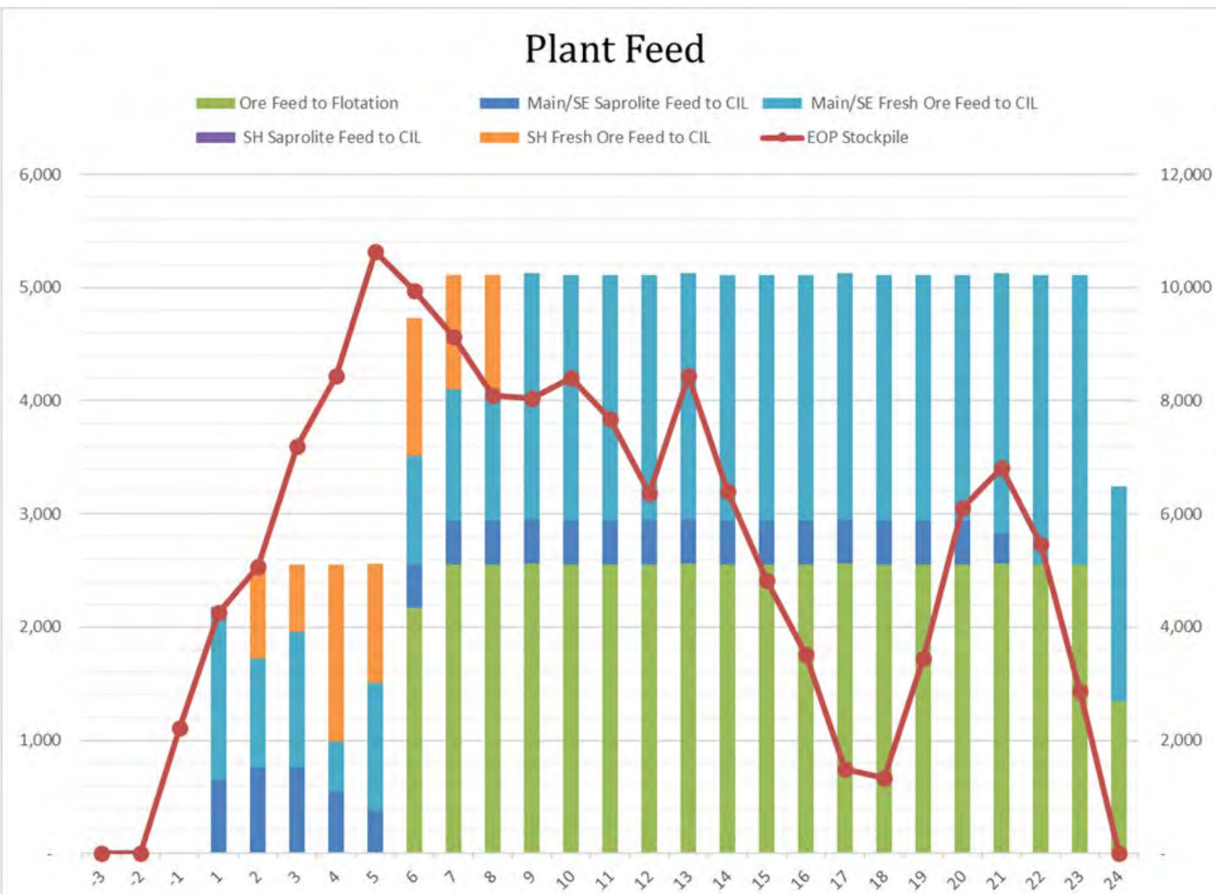
Table 22-3: Plant Rock Feed Summary

Description	Value	Units
Plant Feed		
Main/SE Saprolite Feed to CIL	9,132	kt
Main/SE Fresh Rock Feed to CIL	43,882	kt
SH Saprolite Feed to CIL	0	kt
SH Fresh Rock Feed to CIL	7,307	kt
Fresh Rock Feed to Flotation	46,981	kt
Total New Feed	**107,302**	**kt**
Rougher Scav. to CIL *	815	kt
Cleaner Tail to CIL *	1,319	kt
Plant Feed Au Grades		
Main/SE Saprolite Feed to CIL Au	1.52	g/t
Main/SE Fresh Rock Feed to CIL Au	2.39	g/t
SH Saprolite Feed to CIL Au	-	g/t
SH Fresh Rock Feed to CIL Au	1.61	g/t
Fresh Rock Feed to Flotation Au	1.11	g/t
Rougher Scav to CIL Au	2.29	g/t
Cleaner Tail to CIL Au	4.42	g/t
Plant Feed Cu Grades		
Main/SE Saprolite Feed to CIL Cu	N/A	%
Main/SE Fresh Rock Feed to CIL Cu	N/A	%
SH Saprolite Feed to CIL Cu	N/A	%
SH Fresh Rock Feed to CIL Cu	N/A	%
Fresh Rock Feed to Flotation Cu	0.17%	%
Rougher Scav to CIL Cu	N/A	%

Description	Value	Units
Cleaner Tail to CIL Cu	N/A	%
Plant Feed Ag Grades		
Main/SE Saprolite Feed to CIL Ag	1.22	g/t
Main/SE Fresh Rock Feed to CIL Ag	0.72	g/t
SH Saprolite Feed to CIL Ag	-	g/t
SH Fresh Rock Feed to CIL Ag	0.92	g/t
Fresh Rock Feed to Flotation Ag	1.50	g/t
Rougher Scav to CIL Ag	1.93	g/t
Cleaner Tail to CIL Ag	4.83	g/t

Source: SRK, 2021. * Recirculated Material

Production is supported by a CIL circuit during the first five-year first phase of the Project (Phase 1). The production schedule was developed with extra capacity at the mine which presents more options for plant feeding and enables feeding higher grades first. All material containing high Cu grades or lower Au grades are stockpiled to be fed to the process in the second phase of the Project. Figure 22-2 presents the composition of the plant feed and the material stock size over the LoM and highlights the evolution from Phase 1 to Phase 2.



Source: SRK, 2021

Figure 22-2: Plant feed summary and stockpile size

Table 22-4 and Table 22-5 present the LoM metal quantities fed to the processing circuits and the associated metallurgic recoveries assumed.

Table 22-4: Plant Metal Feed Summary

Description	Value	Units
Plant Feed Au Metal		
Main/SE Saprolite Feed to CIL Au	447	oz
Main/SE Fresh Rock Feed to CIL Au	3,372	oz
SH Saprolite Feed to CIL Au	-	oz
SH Fresh Rock Feed to CIL Au	379	oz
Fresh Rock Feed to Flotation Au	1,682	oz
Rougher Scav to CIL Au	60	oz
Cleaner Tail to CIL Au	187	oz
Plant Feed Cu Metal		
Fresh Rock Feed to Flotation Cu	177,132	lb
Plant Feed Ag Metal		
Main/SE Saprolite Feed to CIL Ag	357	oz
Main/SE Fresh Rock Feed to CIL Ag	1,018	oz
SH Saprolite Feed to CIL Ag	-	oz
SH Fresh Rock Feed to CIL Ag	217	oz
Fresh Rock Feed to Flotation Ag	2,259	oz
Rougher Scav to CIL Ag	50	oz
Cleaner Tail to CIL Ag	205	oz

Source: SRK, 2021

Table 22-5: Plant Metal Recoveries

Description	Value	Units
Plant Feed Au Recoveries		
Main/SE Saprolite Feed to CIL Au	95.5%	%
Main/SE Fresh Rock Feed to CIL Au	92.9%	%
SH Saprolite Feed to CIL Au	0%	%
SH Fresh Rock Feed to CIL Au	82.3%	%
Fresh Rock Feed to Flotation Au	65.7%	%
Rougher Scav to CIL Au	61.9%	%
Cleaner Tail to CIL Au	61.6%	%
Plant Feed Cu Recoveries		
Fresh Rock Feed to Flotation Cu	83.6%	%
Plant Feed Ag Recoveries		
Main/SE Saprolite Feed to CIL Ag	86.8%	%
Main/SE Fresh Rock Feed to CIL Ag	60.0%	%

Description	Value	Units
SH Saprolite Feed to CIL Ag	84.4%	%
SH Fresh Rock Feed to CIL Ag	53.0%	%
Fresh Rock Feed to Flotation Ag	60.5%	%
Rougher Scav to CIL Ag	43.2%	%
Cleaner Tail to CIL Ag	49.7%	%

Source: SRK, 2021

The leaching circuit was designed to recover doré bars containing payable quantities of Au and Ag, while the flotation circuit was designed to recover a Cu concentrate also containing payable quantities of Au and Ag. Table 22-6 presents the LoM product output.

Table 22-6: Project Product Summary

Description	Value	Units
Doré Production		
Au	4,337	oz
Ag	1,302	oz
Concentrate Production		
Au	1,106	oz
Copper	148,141	lb
Ag	1,367	oz

Source: SRK, 2021

Commodity prices used in the calculation of financial results continued within the prefeasibility study are US$1,500 per ounce or Au, US$20.22 per ounce of Ag and US$3.13 per pound of Cu. This study assumes execution of the Precious Metals Purchase Agreement signed with Wheaton Precious Metals Corp. (Wheaton) under which Wheaton purchases 10% of the Au produced over the LoM at US$400/oz of payable Au and 50% of the Ag produced over the LoM at US$3.90/oz payable Ag. As part of the agreement Wheaton contributes US$138 million as an advance deposit at the time of construction of the Project. The Company indicates that the entirety of the US$138m will be provided under this agreement during the pre-production construction period year -1 and -2.

The evaluation considers the following terms for the calculation of doré and Cu concentrate net smelter return.

Doré

- Au Payable 99.95%;
- Ag Payable 99.25%;
- Au Refining Charge US$0.48/oz Au;
- Ag Refining Charge US$0.48/oz Ag; and
- Secured Air Transport and Insurance US$2.45/oz Doré Copper Concentrate.

The Cu concentrate will have a Cu grade of 21% and will yield significant quantities of Au, which could result in a scenario where Au is the major value contributor of these concentrates. It is expected that there will be some bismuth and selenium in the concentrate that will cause the Company to pay penalties.

The cash flow model assumes concentrates will be bulk shipped in containers to Europe, on a weekly or bi-monthly basis. The following are the assumed net smelter return terms.

- Copper Payable — Deduction of 1% point;
- Au Payable — 97% of contained Au;
- Ag Payable — 90% of contained Ag;
- Treatment Charge — US$60.00/t;
- Copper Refining Charge — US$0.060/payable lb Cu;
- Au Refining Charge — US$4.50/ payable oz Au;
- Ag Refining Charge — US$0.45/payable oz Ag;
- Penalties (Se / Bi) — US$5.00 /t; and
- Conc. Transp. and Insurance — 0.167% of payable Metals.

Additional road transportation cost of US$12.15/dmt and a handling charge of US$25.00/dmt of material is assumed due to the characteristics of the material for special treatment, handling, storage etc. (for container shipments). This is in addition to ocean freight charges of US$100/dmt of concentrate.

The model includes the 30% corporate income tax rate of Guyana. The C cash flow in each year of the Project life is discounted back to the end of Year -3 to determine the estimated discounted cash flow at a 5%, 8% and 10% discount rate. Using this same data, the estimated internal rate of return and the undiscounted cash flow were also determined.

The PEA makes use of Potential Mineable Measured, Indicated and Inferred Resources. Inferred Resources represent only about 5% of the total mineable material.

Depreciation – Depreciation of US$1,299 million during the life of the operation includes initial capital of US$355 million, expansion plus sustaining capital of US$944 million.

Startup – For the purpose of the model, the plant is estimated to commence the processing on January 1 of Year 1.

Working capital – Working capital was included in the model. This estimate was considered as 20% of all operating costs for each period.

Taxation – A 30%corporate tax rate was applied over the life of the Project.

Escalation – The components of the economic model were based on the following:

- Base capital pricing for the Project is in Q4 2021 US dollars, with no allowances for inflation or escalation beyond that time;
- Equipment cost estimate from first principles based on equipment cost databases; and
- Operating costs estimates from first principles in Q4 2021 terms.

All financial results are based in Q4 2021, and no escalation has been assumed for the metal prices or cost inputs.

Closure Costs – For the purposes of the financial model, these costs were incurred over a period of one year, following the processing of the last economic material through the mill. No credit was provided in the model for the potential salvage value of equipment.

22.3 Results

Based on the parameters aforementioned, Project evaluation resulting economics present an after-tax net present value of US$794 million, at 5% discount rate, and an internal rate of return of 46.08%. Table 22-7 presents further details of the economic results.

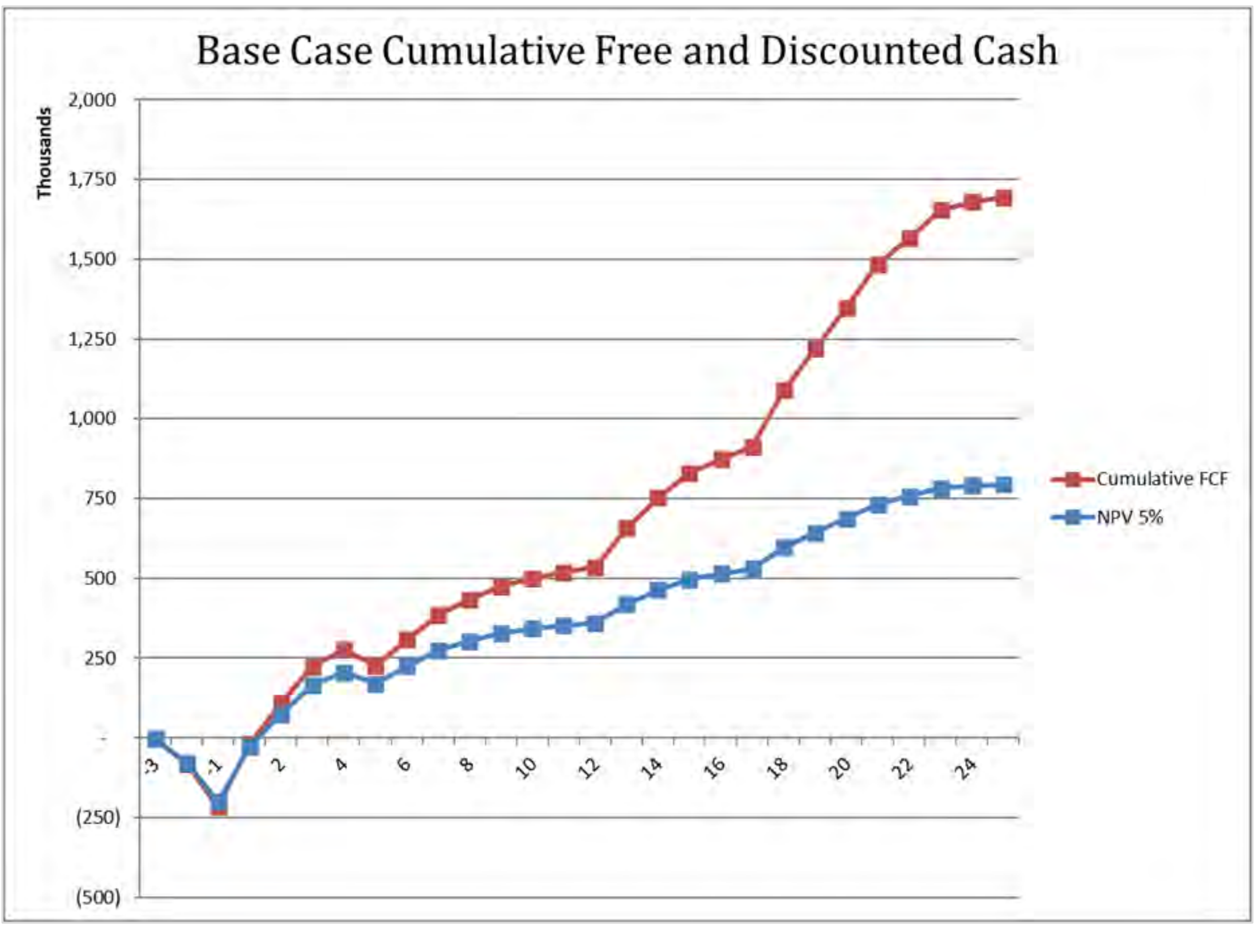
Table 22-7: Project Evaluation Economic Results

Description	Value US$000's
Metal Prices	
Au – Sold to Market (US$/oz)	$1,500
Au – Sold to WPM (US$/oz)	$400.00
Ag – Sold to Market (US$/oz)	$20.22
Ag – Sold to WPM (US$/oz)	$3.90
Copper	$3.13
Estimate Of Cash Flow (All Values in US$000's)	
Gross Income	
Payable Au (Doré+Concentrate)	$7,516,386
Payable Ag (Doré+Concentrate)	$30,413
Payable Copper (US$/lb)	$441,660
Gross Income	**$7,988,460**
Treatment Charges	($18,634)
Refining Charges	($18,019)
Predicted Penalties	($1,553)
Freight Insurance Cost	($63,508)
Gross Revenue	**$7,886,746**
Guyana Au Royalty	($595,750)
Guyana Ag Royalty	($2,018)
Guyana Cu Royalty	($6,185)
One Time Royalty to Surface Owner	($20,000)
Net Revenue	**$7,262,794**
Operating Costs	
Mining Cost	($1,840,872)
Processing Cost	($1,557,511)
Site G&A Cost	($360,096)
Total Operating	**($3,758,479)**
/t ore	($35.03)
Cash Cost (/Au oz)	($695)
Operating Margin (EBITDA)	**$3,504,315**
Initial Capital	($354,760)

Description	Value US$000's
Sustaining Capital	($943,811)
PMPA Installments	$138,000
Income Tax	($649,572)
Free Cash Flow	**$1,694,172**
After-Tax IRR	**46.08%**
After-Tax Present Value 5%	**$794,034**
After-Tax Present Value 8%	**$535,423**
After-Tax Present Value 10%	**$420,676**

Source: SRK, 2021

The base case payback period is estimated at 1.15 years. Figure 22-3 presents the cumulative free and discounted cash flow profile.



Source: SRK, 2021

Figure 22-3: Cumulative free and discounted cash flow

The economic modelling resulted in a LoM cash cost of US$916/Au oz, as presented in Table 22-8.

Table 22-8: Project LoM Cash Cost

PEA Cash Cost Estimates	**LoM Average ($/oz. Payable Gold)**
Mining Cost (open pit & underground)	$340
Processing Cost	$288
Site G&A Cost	$67
Freight & Insurance Cost	$12
Treatment Charges	$3
Refining Charges	$3
Predicted Penalties	$0
By-Product Credit	($87)
Royalties	**$115**
Cash Cost	**$742**
Sustaining Capital	$175
All-in sustaining cash cost (AISC)	**$916**

Source: SRK, 2021

Table 22-9 shows annual production and cash flow forecasts for the life of the Project.

Table 22-9: Project LoM Cash Cost

Years	**Mined Resource (kt)**	**Leaching Feed (kt)**	**Flotation Feed (kt)**	**Payable Au Produced (kozs)**	**Payable Ag Produced (kozs)**	**Payable Cu Produced (klbs)**	**Free Cash US$000's**	**Discounted Cash US$000's**
-3	-	-	-	-	-	-	(3,474)	(3,474)
-2	-	-	-	-	-	-	(78,164)	(74,442)
-1	2,214	-	-	-	-	-	(135,122)	(122,559)
1	4,212	2,178	-	280	70	-	198,092	171,120
2	3,373	2,555	-	228	57	-	125,650	103,373
3	4,684	2,555	-	204	104	-	116,407	91,208
4	3,796	2,555	-	207	63	-	51,782	38,641
5	4,757	2,562	-	189	24	-	(47,837)	(33,997)
6	4,036	2,555	2,172	171	152	8,012	80,820	54,702
7	4,300	2,555	2,555	165	212	9,628	77,413	49,901
8	4,066	2,555	2,555	160	136	7,695	47,569	29,203
9	5,080	2,562	2,562	213	163	8,670	41,343	24,172
10	5,480	2,555	2,555	203	179	9,696	25,152	14,006
11	4,369	2,555	2,555	198	97	10,521	19,362	10,268

Years	Mined Resource (kt)	Leaching Feed (kt)	Flotation Feed (kt)	Payable Au Produced (kozs)	Payable Ag Produced (kozs)	Payable Cu Produced (klbs)	Free Cash US$000's	Discounted Cash US$000's
12	3,812	2,555	2,555	225	77	7,576	15,170	7,662
13	7,199	2,562	2,562	290	155	11,680	123,543	59,427
14	3,065	2,555	2,555	248	124	6,604	94,551	43,315
15	3,541	2,555	2,555	245	74	4,357	78,162	34,102
16	3,787	2,555	2,555	214	83	6,597	42,581	17,693
17	3,111	2,562	2,562	232	58	6,907	40,122	15,878
18	4,942	2,555	2,555	340	91	5,749	175,891	66,292
19	7,227	2,555	2,555	352	124	7,336	130,818	46,956
20	7,763	2,555	2,555	275	140	7,265	127,168	43,472
21	5,832	2,562	2,562	300	115	7,523	137,772	44,854
22	3,767	2,555	2,555	203	98	9,321	80,626	25,000
23	2,523	2,555	2,555	173	56	4,404	88,178	26,039
24	368	1,899	1,346	92	72	1,754	26,155	7,356
25	-	-	-	-	-	-	14,442	3,868
26	-	-	-	-	-	-	-	-
Total	**107,302**	**60,322**	**46,981**	**5,407**	**2,522**	**141,295**	**1,694,172**	**794,034**

Source: SRK, 2021

22.4 Sensitivity Analysis

Sensitivities were run considering the variation of capital and operating costs and also to metal prices. Table 22-10 to Table 22-12 and Figure 22-4 present the results of this sensitivity analysis.

Table 22-10: Capital Cost Sensitivity

Capital Costs Sensitivity	80%	90%	100%	110%	120%
After-Tax NPV 5%	922,979	858,511	794,034	729,423	664,796
After-Tax NPV 8%	644,866	590,150	535,423	480,525	425,607
After-Tax NPV 10%	520,254	470,471	420,676	370,696	320,695
IRR	73.25%	57.57%	46.08%	37.30%	30.48%

Source: SRK, 2021

Table 22-11: Operating Cost Sensitivity

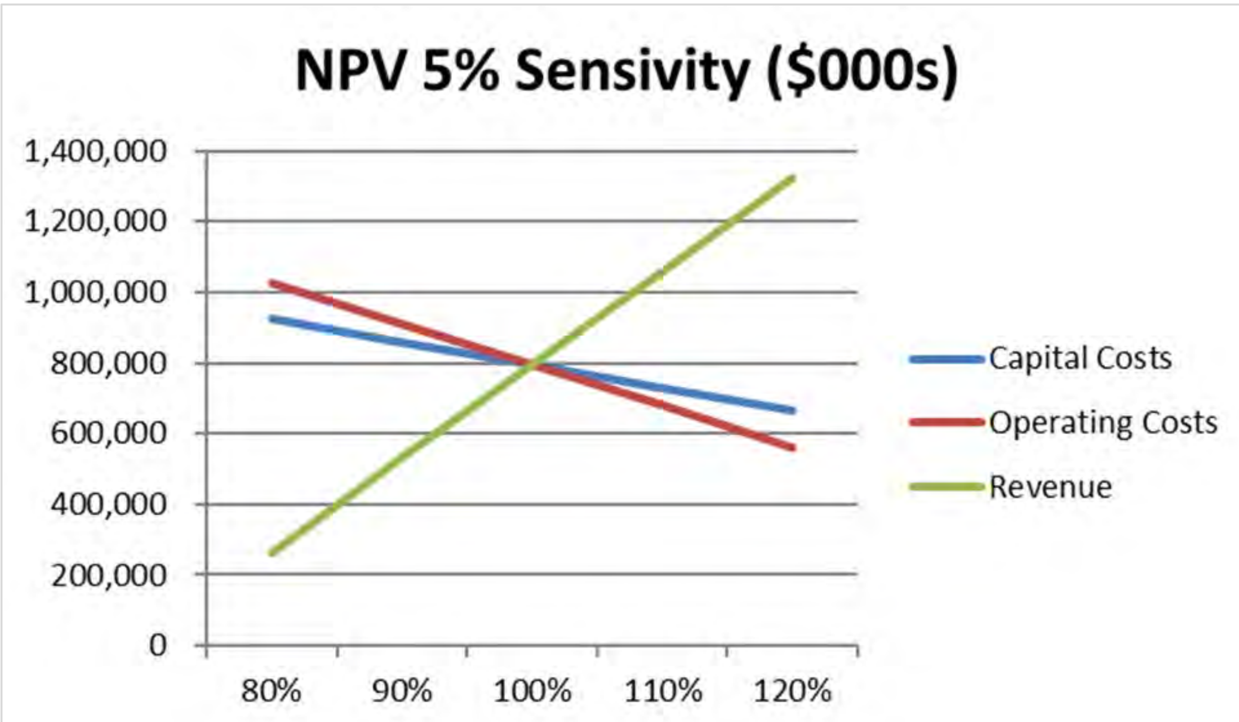
Operating Costs Sensitivity	80%	90%	100%	110%	120%
After-Tax NPV 5%	1,026,482	910,263	794,034	677,714	561,294
After-Tax NPV 8%	696,770	616,103	535,423	454,627	373,714
After-Tax NPV 10%	550,048	485,368	420,676	355,857	290,919
IRR	52.24%	49.24%	46.08%	42.66%	38.94%

Source: SRK, 2021

Table 22-12: Metal Price Sensitivity

Revenue Sensitivity	80%	90%	100%	110%	120%
After-Tax NPV 5%	259,827	529,132	794,034	1,058,536	1,323,028
After-Tax NPV 8%	152,083	346,232	535,423	724,115	912,795
After-Tax NPV 10%	106,232	265,925	420,676	574,895	729,102
IRR	21.22%	35.25%	46.08%	55.46%	64.00%

Source: SRK, 2021



Source: SRK, 2021

Figure 22-4: Sensitivity spider graph

23 ADJACENT PROPERTIES

Nordmin is not aware of any significant properties situated immediately adjacent to the Project.

24 OTHER RELEVANT DATA AND INFORMATION

There is no other additional information or explanation necessary to make the Technical Report understandable and not misleading.

25 INTERPRETATION AND CONCLUSIONS

25.1 Introduction

The QP's note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this Technical Report.

25.2 Mineral Tenure, Surface Rights, Royalties, and Agreements

The Project is located within the Company's 100% controlled Upper Puruni Concession contains 53,283-hectare(s) of mineral leases located in the Cuyuni–Mazaruni Region (Region 7) of Western Guyana, South America (referred to as "the Property"). The Project is comprised of two deposits: the Toroparu Deposit (formerly known as Toroparu or Toroparu Main and Toroparu SE), and the Sona Hill Deposit (formerly known as Sona Hill, Sona Hill prospect, and Sona Hill Gold Deposit). The Toroparu Deposit is located near the main camp. The Sona Hill Deposit is located approximately 5 km to the southeast of the Toroparu Deposit.

The Toroparu Deposit, Main and SE Areas, are located on property subject to the Alphonso Joint Venture. The Sona Hill Deposit is located on property that is subject to the Godette Joint Venture.

ETK has all the necessary permits and permissions currently required to conduct its exploration work and medium-scale mining and gravity recovery of gold and other minerals on the Project. In addition, the project has its Environmental Authorization, Mineral Agreement, and Fiscal Stability Agreement in place.

The Company executed a Mineral Agreement with the Government of Guyana (the "Mineral Agreement") that stipulates a royalty of 8% on gold (1.5% on copper) produced from its mineral properties payable in cash or in-kind to the Government of Guyana.

Mineral properties are also subject to annual rentals. The rental rates for each of the MPs are US$1.00 per acre per annum. Rental rates for each of the PPMS are US$0.25 per acre for the first year, with an increment of US$0.10 per acre for every additional year. Rental rates for PLs are US$0.50 per acre for the first year, US$0.60 per acre for the second year, and US$1.00 per acre for the third year with an increase of $0.50 per acre for the fourth and fifth years.

25.3 Exploration, Drilling, and Analytical Data Collection in Support of Mineral Resource Estimation

Until the beginning of 2011, the Upper Puruni Concession package (1,000 km^2) had remained unexplored. A systematic surface sampling and mapping approach was implemented starting in 2011, focused primarily on geological potential for gold and/or base metals. Targets were originally selected from interpretations of airborne geophysical data and sat imagery. In areas where geochemical sampling yielded positive results, tighter grid spacing for ground geophysics was carried out.

Geochemical samples were taken from the soil layer, and if possible, the laterite layer, averaging 0.5 m to 0.3 m depth. This was done using a hand auger. The geochemical sampling resulted in identifying the Toroparu Deposit NW area, the Ameeba hills geochemical anomaly (a possible extension of the Toroparu Deposit) that led to follow up DDH drilling.

Geochemical sampling in 2012, added 3,251 samples. Sampling during this program confirmed three new anomalies, Sona Hill, Sona Hill South, and Majuba located south and southeast of the Toroparu Deposit.

Drilling has occurred at the Project from 2006 through to 2021, directed primarily at the Main and SE Areas of the Toroparu Deposit. At the end of 2021 over 14 years, a total of 215,154 m of resource definition drilling was completed in 528 holes. Since 2013 most of the exploration drilling throughout the property has been directed toward other exploration targets such as the Sona Hill and Wynamu target areas. Sona Hill was drilled from late 2015 to early 2018 with 181 diamond drill holes and 20,850 m of drilling: sufficient for resource estimation. Wynamu was drilled with 62 core holes for 6,432.6 m of drilling.

The November 2021 Mineral Resource Estimate was prepared by Nordmin following a two-phase diamond drill program in 2020-2021 which comprised a total of 20,750 m in 114 drill holes. Previously the deposit was modelled as a large low-grade, high tonnage system. Updates identified cross-cutting high-grade structures throughout the deposit, drilling intersected multiple intervals of VG, confirming updated modelling.

The quantity and the quality of lithological, collar, and downhole survey data collected in the various exploration programs by various operators are sufficient to support the Mineral Resource Estimate. The collected sampling is representative of gold, total copper, cyanide soluble copper, and silver data in the deposits, reflecting areas of higher, and lower grades. The analytical laboratories used for legacy and current assaying are well known in the industry, produce reliable data, are properly accredited, and are widely used within the industry.

Nordmin is not aware of any drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results. In Nordmin's opinion the drilling, core handling, logging, and sampling procedures meet or exceed industry standards, and are adequate for the purpose of Mineral Resource Estimation.

Nordmin considers the QA/QC protocols in place for the Project to be acceptable and in line with standard industry practice. Based on the data validation and the results of the standard, blank, and duplicate analyses, Nordmin is of the opinion that the assay and SG databases are of sufficient quality for Mineral Resource Estimation for the Project.

25.4 Geology and Mineralization

The Guiana Shield, the northern half of the Amazonian Craton, underlies the eastern part of Venezuela, Guyana, Surinam, French Guyana, and parts of northern Brazil. It is also among the least documented of Precambrian terranes due to thick weathering profiles, tropical vegetation, and tertiary sands (Voicu, Bardoux, & Stevenson, 2001). This region is bound in the north by the Atlantic Ocean and the south by the Amazon-Solimoes basin. There are two undisputed terranes in the Guiana Shield, the Imataca Complex in northwestern Venezuela and the Trans-Amazonian granitoid-greenstone belts in the easternmost extension in Amapá, Brazil. The Toroparu Deposit is located close to and between two major lineaments; the WNW oriented Puruni fault zone, to the southwest, and the NNW striking Wynamu fault, likely affecting the southeast portion of the deposit. Within such a regional structural pattern the mineralized zones of the Toroparu Deposit can be interpreted as east-west oriented, west plunging, dilational zones within an WNW oriented, oblique sinistral strike-slip fault zone. More structural evidence is needed to fully support this interpretation of higher-grade E-W lenses within the overall WNW oriented orebody.

The Toroparu Deposit mineralization is oriented in a west-northwest direction with cross-cutting east-west mineralized structures. The system corresponds to a 2.7 km long and 200 m to 40 m wide body and extends to over 400 m in depth. The mineralized body occurs along the northwestern boundary of a tonalitic to quartz dioritic intrusion within a series of mafic volcanics with a thick, gradational layer of saprolite material. Saprolite results from deep tropical weathering, resulting in the larger part of the original rock mineralogy being replaced by clays. Quartz veins and veinlet networks survive quite well in saprolite and contain occasional free gold grains. Sulphides tend to be completely leached and removed, leaving relic voids, and/or oxidized spots. The Toroparu Deposit sits in a topographic low and is near the Puruni and Wynamu rivers. This has resulted in the upper part of the lateritic profile being eroded. Bedrock substratum is overlain by a thin, 1 m residual soil layer, followed by a 10 m 35 m thick saprolite layer. Saprolite rock is the transitional zone between saprolite and fresh rock, creating a gradational contact several metres thick at the Toroparu Deposit.

Within the Toroparu Deposit, mineralization is hosted by a paleoproterozoic greenschist facies metamorphic VS sequence in contact with a tonalitic to quartz dioritic intrusives. Gold and copper mineralization appears to be largely controlled by a series of moderately developed, dilational brittle-ductile fracture veinlet stockworks. This dilational fracture veinlet stockwork forms a NW-SE trending mineralized corridor with two sets of cross-cutting higher-grade structures. Where these structures intersect, there is a large increase in the grade of both gold and copper. There is also either massive veining or vein breccias in these intersections. The Main Area contains the majority of known mineralization, which is still open at depth. The northwestern lens of the Main Area appears to have slightly lower concentrations of gold grades, but this could be due to a lack of drill density; mineralization here is also open along strike to the NW and at depth.

Sona Hill Deposit differs from the Toroparu Deposit in the absence of potentially economic quantities of copper mineralization. Gold mineralization is hosted in sub horizontal, shallow dipping structures. It has two sets of identified cross-cutting high-grade gold structures. The Sona Hill saprolite is generally thicker, as the 25 m to 30 m of topographic hill results in a greater depth to the water table. Sap-rock and saprolite layers can reach up to 60 m thick in the Sona Hill Deposit.

Similar to the Toroparu Deposit, Main, and SE Areas, mineralization at the Sona Hill Deposit is mainly hosted within intrusive lithologies. These intrusives are petrographically described as porphyritic/micro-porphyritic ± equigranular granodiorite to quartz diorite. Metavolcanics are foliated andesitic volcaniclastics and intermediate to felsic flows. Quartz veining is typically white-crystalline quartz, and can be associated with feldspar, carbonate, tourmaline, sericite, and chlorite with minor sulphides (pyrite). Veins/veinlets are variable in size but generally range from 0.5 cm to 10 cm, density varies significantly. Alteration is quartz-sericite-carbonate-chlorite which is both pervasive throughout the deposit and present as vein halos.

The Toroparu and Sona Hill Deposits are a part of a single coherent structural system related to thrusting that carries hanging wall blocks eastward over footwall blocks. Mineralization here is hosted within the frontal part of the back-thrust zone.

25.5 Metallurgy, Processing and Recoveries

Comprehensive metallurgical test work programs were conducted on the Toroparu Deposit saprolite and fresh rock Au-bearing material by Inspectorate Exploration and Mining Services Ltd. of Richmond, British Columbia (BC) (2012-2013); SGS Canada Inc. of Lakefield, Ontario (2009-2013); ALS of Kamloops, BC (2013-2014), and FLSmidth Dawson Metallurgical Laboratory of Salt Lake City, Utah (2014).

The Sona Hill saprolite and fresh rock test work was performed by Base Metal Laboratories of Kamloops, BC (2019-2020). Testwork included comminution, gravity concentration, flotation, and cyanidation for metallurgical recovery, as well reagent consumptions for the various rock types identified during previous engineering studies.

Test work demonstrated that multiple processes are necessary to provide economic benefit to the different mineralized material in the deposit. Metallurgical test work studies were performed to show that processing the deposit with both flotation and cyanide leaching, depending on Cu content, would provide economic benefit due to the recovery of a marketable Cu concentrate.

Process facilities were designed to achieve the stated recoveries based on test results and standard engineering design practices. Process facilities include comminution circuits consisting of primary crushing and SAG milling for the Gold Plant and a stand alone three-stage crushing circuit feeding ball mill grinding circuit for the Flotation Plant. The Gold Plant includes gravity process as part of the comminution circuit followed by cyanide leaching via the CIL process. Tailings from the Gold Plant will be treated through a cyanide detoxification circuit prior to discharge into the TSF facility. The Flotation Plant comprises rougher and cleaner flotation stages followed by concentrate thickening and filtration. Flotation tails are thickened and combined with the Gold Plant tailings. Products from the process facility include Au doré.

25.6 Mineral Resource Estimate

The Mineral Resource Estimate for the Project conforms to industry best practices and is reported using the 2014 CIM Definition Standard for Mineral Resources and Mineral Reserves and 2019 CIM Best Practice Guidelines. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. This estimate of Mineral Resources may be materially affected by environmental permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

The Mineral Resource Estimate was calculated from two main databases for the Project, one for the Toroparu Deposit and another for the Sona Hill Deposit. Both complete databases are comprised of a total of 709 diamond drill holes and three trenches consisting of 199,996 m. This includes:

- Toroparu Deposit has 528 diamond drill holes consisting of 178,491 m and three trenches comprised of 655.3 m completed between 2006 and 2021, and
- Sona Hill Deposit area has 181 diamond drill holes consisting of 20,850 m completed between 2012 and 2018.

The November 2021 Mineral Resource Estimate was prepared by Nordmin following a two-phase diamond drill program in 2020-2021 which comprised a total of 20,750 m in 114 drill holes. The new drill hole assays were reviewed and fully validated by Nordmin.

Nordmin, through an interactive process with the Company, undertook a full re-examination of the mineralogical, lithological, structural, and geochemical correlations influencing gold mineralization within the Project. The review concluded that:

- The previous modelling of the mineralization utilizing a single implicit lower grade 0.2 g/t gold shell did not identify nor isolate the structurally controlled higher-grade domains that exist throughout the project area.
- The previous interpretation was not representative of the deposit type nor the geological controls of mineralization that support both lower grade and higher-grade mineralized domains.
- Each domain and corresponding sub domains required extensive modelling of the higher-grade structural domains, which control the higher-grade mineralization within the encapsulating lower grade mineralized domain.

The 2020 and 2021 20,750 m (114 hole) drill program further verified the location and structural relationship between the lower and higher-grade mineralization domains located within the previously defined disseminated lower grade mineralized halo along the 4 km Toroparu trend and for the Sona Hill Deposit. Nordmin incorporated the various geological, structural controls to support the various gold, copper, and silver mineralization styles, and their associated geochemistry. The block model utilized explicit modelling of mineralized structures present in the deposit areas to support the Mineral Resource Estimate. These models incorporate the geologic and structural controls of gold mineralization, the style of mineralization, and its associated geochemistry. The Toroparu Deposit consists of multiple geographical areas, including the Main, NW, and SE Areas. Each of these areas was separated into various domains. The Sona Hill Deposit used three main domains for the estimation process.

The intersection of the NW-SE and E-W structures creates zones of wider and higher-grade gold mineralization than in the structures themselves. These structural intersections occur over a consistent and repeatable pattern that enriches gold, silver, and copper mineralization throughout the deposits. The recognition of these patterns supports the combination of open pit and underground mining methods that form the basis of the Mineral Resource Estimate.

The Mineral Resource was classified in accordance with the 2014 CIM Definition Standards and 2019 CIM Best Practice Guidelines. Mineral Resource classifications or “categories” were assigned to regions of the block model based on the QP’s confidence and judgment related to geological understanding, continuity of mineralization in conjunction with data quality, spatial continuity based on variography, estimation pass, data density, and block model representativeness, specifically assay spacing and abundance, kriging variance, and search volume block estimation assignment.

For the Toroparu Deposit, the classification was initially applied from the estimation pass. Blocks populated in pass 1 were classified as Measured, blocks populated in pass 2 were classified as Indicated, and blocks populated in pass 3 were classified as Inferred. Subsequently, the block model was analyzed, and it was determined that classification adjustments were required depending on the drilling density required to support an underground or an open pit resource; blocks in the first, second, and third pass that display a relatively high kriging variance were downgraded to a lower classification. For the Sona Hill Deposit, classification was applied directly from the estimation pass. Blocks populated in pass 1 were classified as Measured, blocks populated in pass 2 were classified as Indicated, and blocks populated in pass 3 were classified as Inferred.

The Mineral Resource Estimate, which is summarized in Table 25-1 and Table 25-2. The updated Mineral Resource Estimate includes an open pit and a maiden underground resource estimate within the Toroparu Main & NW and SE deposits along with the satellite deposits consisting of the Southeast zone (SE) and the Sona Hill satellite gold deposits.

Table 25-1: Mineral Resource Statement for the Toroparu Project

Deposit	Area	Resource Category	Type	Tonnes ('000s)	Au (g/t)	Au oz ('000s)	Cu (%)	Cu lb ('000s)	Ag (g/t)	Ag oz ('000s)
Toroparu	**Main/NW**	Measured	Open pit	98,070	1.21	3,809	0.110	238,112	1.19	3,743
		Indicated		62,531	1.56	3,133	0.100	137,557	0.91	1,828
Toroparu	**SE**	Measured	Open pit	5,121	1.16	190	0.043	4,826	n/a	n/a
		Indicated		2,403	1.14	88	0.052	2,763	n/a	n/a
Sona Hill	**Sona Hill**	Measured	Open pit	6,958	1.85	413	0.008	1,241	1.07	239
		Indicated		4,180	1.66	223	0.008	700	0.85	115
Toroparu	**Main/NW**	Measured	Underground	727	2.84	66	0.072	1,151	0.47	11
		Indicated		4,978	3.21	514	0.091	9,937	0.41	66
Total Measured				**110,877**	**1.26**	**4,479**	**0.100**	**245,330**	**1.12**	**3,993**
Total Indicated				**74,092**	**1.66**	**3,958**	**0.092**	**150,957**	**0.84**	**2,009**
Total Measured & Indicated				**184,969**	**1.42**	**8,437**	**0.097**	**396,286**	**1.01**	**6,002**
Toroparu	**Main/NW**	Inferred	Open Pit	4,018	1.58	204	0.080	7,118	0.66	85
Toroparu	**SE**	Inferred	Open Pit	9	1.67	1	0.040	8	n/a	n/a
Sona Hill	**Sona Hill**	Inferred	Open Pit	1,365	1.28	56	0.006	179	0.54	24
Toroparu	**Main/NW/SE**	Inferred	Underground	8,403	3.53	953	0.091	16,884	0.25	68
Total Inferred				**13,796**	**2.74**	**1,213**	**0.08**	**24,189**	**0.40**	**177**

Table 25-2: Mineral Resource Estimate Summary

	Tonnes ('000s)	Au (g/t)	Au oz ('000s)	Cu (%)	Cu lb ('000s)	Ag (g/t)	Ag oz ('000s)
Open Pit							
Measured and Indicated	179,264	1.36	7,857	0.097	385,198	1.03	5,924
Inferred	5,393	1.50	260	0.061	7,305	0.63	109
Underground							
Measured and Indicated	5,705	3.16	580	0.088	11,088	0.42	77
Inferred	8,403	3.53	953	0.091	16,884	0.25	68
Total							
Measured and Indicated	184,969	1.42	8,437	0.097	396,286	1.01	6,002
Inferred	13,796	2.74	1,213	0.080	24,189	0.40	177

Mineral Resource Estimate Notes

1. Combined Open Pit and Underground Mineral Resources were prepared in accordance with NI 43-101 and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. This estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
2. Underground and Open Pit Mineral Resources are based on a gold price of $1,630/oz. This gold price is the three-year tailing average as of September 30, 2021.
3. Open Pit Mineral Resources comprise the material contained within various Lerchs-Grossmann pit shells at various revenue factors. These revenue factors are as follows: Main/Southeast/NW Zone @ 0.75 revenue factor and Sona Hill @ 1.00 revenue factor. The gold cut-off applied to Open Pit Mineral Resources within the selected pit shells was 0.40 g/t.
4. Underground Mineral Resources comprise all material found within MSO wireframes generated at a cut-off of 1.8 g/t gold including material below cut-off.
5. Silver values are not reported for the SE Open Pit Ag contained metal values reported will not equal A tonnes X grade conversion calculation.
6. Assays were variably capped on a wireframe-by-wireframe basis.
7. Specific Gravity was applied using weighted averages to each individual lithology type.
8. Mineral Resource effective date November 1, 2021.
9. All figures are rounded to reflect the relative accuracy of the estimates and totals may not add correctly.
10. Excludes unclassified mineralization located within mined out areas.
11. Reported from within a mineralization envelope accounting for mineral continuity.

Areas of uncertainty that may materially impact the Mineral Resource Estimate include:

- Changes to long term metal price assumptions.
- Changes to the input values for mining, processing, and G&A costs to constrain the estimate.
- Changes to local interpretations of mineralization geometry and continuity of mineralized zones.
- Changes to the density values applied to the mineralized zones.
- Changes to metallurgical recovery assumptions.
- Changes in assumptions of marketability of the final product.
- Variations in geotechnical, hydrogeological, and mining assumptions.
- Changes to assumptions with an existing agreement or new agreements.
- Changes to environmental, permitting, and social license assumptions.
- Logistics of securing and moving adequate services, labour, and supplies could be affected by epidemics, pandemics, and other public health crises, including COVID-19, or similar such viruses.

25.7 Mining Methods

25.7.1 Mineral Resources within the PEA Mine Plan Estimate Summary

The estimate of mineral resources within the PEA mine plan is effective as of December 1, 2021 and is presented in Table 25-3. The PEA models an open pit and an underground mine with mineral resources within the PEA mine plan containing 6.156 Moz of Au, 3.993 Moz of Ag and 240.2 Mlb of Cu (109.0 kt).

Measured, Indicated and Inferred resources were used for conversion to mineral resources within the PEA mine plan within the PEA open pit and underground designs. The open pit mineral resources within the PEA mine plan are contained within the Toroparu Pit, Sona Hill Pit and SE Pit and are associated with 558 Mt of waste and a LoM stripping ratio of 5.99:1. The underground mineral resources within the PEA mine plan are contained below the Toroparu Pit.

The mineral resources within the PEA mine plan are valid at the time of estimation and include CoG assumptions made before the final PEA cash flow model was completed. SRK and Nordmin confirmed the overall project economics are favorable at the approximate four-year moving average Au price of US$1,500/oz Au, an average Ag price of US$20/oz Ag, and an average Cu price of US$3.13/lb Cu.

Table 25-3: Mineral Resources within the PEA Mine Plan

MINERAL RESOURCES WITHIN THE PEA MINE PLAN								
Area	**Resource Category**	**Tonnes ('000s)**	**Au g/t**	**Ag g/t**	**Cu %**	**Contained Au Toz ('000s)**	**Contained Ag Toz ('000s)**	**Contained Cu Tonnes ('000s)**
All Open Pits	Measured	60,117	1.41	1.36	0.11	2,728	2,633	64.6
	Indicated	31,407	1.74	1.12	0.09	1,756	1,126	29.8
	Measured & Indicated	91,525	1.53	1.28	0.10	4,499	3,769	94.5
	Inferred	1,593	1.62	0.89	0.07	83	45	1.1
	All Open Pits Subtotal	**93,118**	**1.53**	**1.27**	**0.10**	**4,567**	**3,804**	**95.5**
Underground	Measured	839	2.73	0.63	0.07	74	17	0.6
	Indicated	5,899	3.24	0.49	0.11	614	92	6.2
	Measured & Indicated	6,738	3.17	0.51	0.10	687	110	6.8
	Inferred	7,447	3.77	0.33	0.09	902	80	6.6
	Underground Subtotal	**14,185**	**3.48**	**0.41**	**0.09**	**1,589**	**189**	**13.4**
All Open Pits & Underground	Measured	60,956	1.43	1.35	0.11	2,802	2,650	65.3
	Indicated	37,306	1.98	1.02	0.10	2,369	1,219	36.0
	Measured & Indicated	98,262	1.64	1.23	0.10	5,187	3,878	101.3
	Inferred	9,040	3.39	0.43	0.09	985	125	7.7
	Grand Total	**107,302**	**1.78**	**1.16**	**0.10**	**6,156**	**3,993**	**109.0**

Source: SRK, 2021 & Nordmin, 2021

<u>Mineral resources within the PEA mine plan estimate notes</u>

- Open Pit Mineral Resources within the PEA Mine Plan:
  - The open pit mineral resources within the PEA mine plan are based on a block by block net smelter return calculation based on an Au price of US$1,500/oz, Ag price of US$20.00/oz and Cu price of US$3.13/lb. The PEA cash flow base case used an Au price of US$1,500/oz., Ag price of US$20.20/oz and Cu price of US$3.13/lb;
  - The open pit mineral resources within the PEA mine plan assume complete mine recovery;
  - The open pit mineral resources within the PEA mine plan are diluted at approximately 15-30% (further to dilution inherent in the resource model and assumes selective mining unit of 5 m x 5 m x 5 m for Main and NW Pits and 2.5 m x 2.5 m x 5m for Sona Hill and SE pits);
  - Contained in situ gold ounces do not include metallurgical ACO recoveries of 83.6% Cu and 80.2% Au and gold LCO recoveries of 92.2%;
  - Waste tonnes within the open pit is 558 Mt at a strip ratio of 5.99:1 (waste to ore);

    - Costs assumptions are: Mining Costs = US$2.30/t moved, Processing/Tailings Costs = US$15.50/t processed, G&A Costs = $5.95/t processed;
    - An open pit CoG of 0.5 g/t-Au saprolite and 0.5 g/t-Au fresh rock was applied to open pit resources constrained by the ultimate pit design; and
    - The mineral resources within the PEA mine plan estimate for the Project was calculated by Fernando P. Rodrigues, BSc, MBA MMSAQP #01405QP of SRK Consulting, Inc. in accordance with the Canadian Securities Administrators National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and generally accepted Canadian Institute of Mining, Metallurgical and Petroleum "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines ("CIM Guidelines").
- Underground Mineral Resources within the PEA Mine Plan:
    - The mineral resources within the PEA mine plan were prepared by B. Wissent, BEng of Nordmin Engineering Ltd., in accordance with NI 43-101 and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. This estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues;
    - Mineral resources within the PEA mine plan are based on selected MSO wireframes generated at Au cut-off of 2.0 g/t based on an Au price of US$1,500/oz. A small amount of the underground mineral resources within the PEA mine plan is based on material from development with a marginal Au diluted cut-off of 1.25 g/t. The PEA cash flow base case used an Au price of US$1,500/oz., Ag price of US$20.20/oz and Cu price of US$3.13/lb;
    - Underground mineral resources within the PEA mine plan assumes mining recovery at approximately 80% to 92.5% for LHOS and 100% for development;
    - Contained in situ gold ounces do not include metallurgical recoveries;
    - Underground mineral resources within the PEA mine plan are diluted at approximately 12% for LHOS and 5% for development; and
    - Costs assumptions are: Mining Costs = US$36.00/t processed, Processing/Tailings Costs = US$15.50/t processed, G&A Costs = US$6.00/t processed, Operating Cost Marginal Allowance (10%) = US$5.80/t processed.
- Mineral resources within the PEA mine plan tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding;
- "g/t" = gram per metric tonne, "Toz" = troy ounces; and
- Mineral resources within the PEA mine plan effective date: December 1, 2021.

25.7.2 Open Pit Mining

A conventional truck-shovel method was considered for the open pit portion of the Toroparu Deposit. The open pit analysis results in several distinct open pits coalescing into the NW and Main Toroparu Pits over time. The Sona Hill and Southeast Zone (SE) will be developed in a similar fashion beginning in year 3 and 6 respectively. The final dimensions of the NW Pit are approximately 990 m long x 690 m wide x 360 m deep. The dimensions of the Main Pit are approximately 1,300 m long x 750 m wide x 470 m deep. The open pit LoM plan proposes to mine approximately 93 Mt at a cut-off grade of 0.5 g/t Au and 558 Mt of waste rock material. The average stripping ratio for the open pit operations is 6:1 over the LoM. Each pit is currently planned to be developed with 29 phases each. Compacted saprolitic waste material will be used to construct haul roads, facility pads and flood control berms, levies, and other structures.

25.7.3 Underground Mining

Underground development will commence at the beginning of the ninth year of open pit operation and targets 3,500 tpd, ramping up to full production over an approximately two-year period. The ramp-up allows for the main ramp system development at the 250 m elevation down from the surface portal in the NW Pit and to connect to both fresh air and return air raises, providing ventilation and secondary egress for the mine. Underground production is scheduled based on approximately 3,500 tpd mill feed and 750 tpd average waste, excavated using a fleet of 15 and 10 tonne load-haul-dump loaders, hauled with 45 tonne trucks using the ramps to portals entrances and rehandled using the surface fleet. Production is expected to commence in the central area between the Main and NW Pits from 360 Level (approximately 360 m elevation below surface) and continues for the first 2 years in a bottom-up sequence. It is anticipated that mining next transitions to production from lower mining areas below and around Main and NW Pits for approximately the final 10 years of the LoM.

The underground mineralization was evaluated using Deswik's MSO tool to create the mineable inventory. The mining cut-off for the MSO underground inventory was generated based on a 2.0 g/t gold cut-off grade (insitu), which approximately equates to a 1.6 g/t gold mill feed grade. A 1.25 g/t incremental mill feed cut-off grade was selected to apply to development. Stopes were created on 30 m level spacing and a maximum of 15 m length, with an average mineralized width of approximately 7 m. Stopes are mined via longitudinal retreat and are accessed by overcut and undercut stope access drifts which extend from the level haulages. The LoM underground mill feed is approximately 14.18 Mt at an average gold grade of 3.48 g/t, and 3.49 Mt of waste.

This PEA is preliminary in nature. In addition to the Measured and Indicated Resources, the mine plan presented in this section includes Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that this PEA will be realized.

25.8 Recovery Methods

The concentrator is designed to process 14,000 tpd of mineralized material (nominal) during its peak operation. The processing plant will be constructed in two phases. The first phase consists of the initial 5 years of the Project where the plant will receive LCO and saprolitic material to recover Au. During the second phase, the plant will be expanded with the addition of a Cu flotation circuit and the associated equipment to produce a Cu concentrate. The overall plant capacity will double in size to 14,000 tpd with the addition of the flotation circuit.

Phase 1 processes 7,000 tpd of LCO and saprolite material through crushing and grinding, CIL circuit and ADR to produce Au doré. This phase continues through the LoM.

In Phase 2, ACO will be of processed at 7,000 tpd of ACO through flotation with cyanide leaching of the cleaner scavenger flotation tailings via a CIL circuit. Based on metallurgical test work recovery by flotation, a Cu concentrate with grade of approximately 21% Cu is expected to be produced.

Gravity concentration with intense cyanidation is performed on a portion of the underflow from the grinding cyclones in both the Gold and Flotation Plants.

25.9 Infrastructure

The project infrastructure design includes a port, road access to site, an airstrip, and camp for employees. Site infrastructure including access and haul roads, tailings pipeline corridor/service road, tailings facility dams, berms, and spillways are designed. Water systems including process makeup water, surface water control, potable water, and firewater are in the design documents. The Project has included the appropriate facilities for operations. The energy needs for the Project include an electric power generation system. A cost estimate for installation of the infrastructure has been included at PEA level in this study.

The TSF, located on the northeast side of the mine property, will be organized and operated in the area covered by Module 1 and Module 3, and has been designed for a storage capacity of 107 Mt (expandable to 156 Mt if required) of slurry tailings. Tailings will be confined by site topography and construction of saddle dams, the typical section being compacted saprolite shells with a chimney drain to relieve the pond head in the center of the final dam and convey seepage through downstream finger drains.

Water balance estimates indicate excess water volumes (mainly due to precipitation) may be discharged into the site area wetland during operations. Water management includes the use of diversion channels, berms and levees to direct all site water to a single wetland for settlement of solids and contaminants prior to discharge of excess water volumes to the environment through a single operating spillway designed for the PMF.

25.10 Environmental Studies, Permitting and Social Impact

The Project area has been historically impacted by mining activities, logging, and hunting. With only a few exceptions, species classified as rare, threatened, or endangered have not been observed in the Project area.

There are no formal or established communities in the immediate vicinity of the site. The Project is not expected to generate many direct socio-economic impacts. A Social Management Plan has been proposed to mitigate the socio-cultural impacts identified in the EIA. No indigenous hunting activity or cultural resources were identified within the proposed mining area. This is not expected to vary based on the PEA mine expansion.

Results of the geochemical testing of the waste rock showed that the waste rock lithologies and low-grade economic material samples contained very low sulphide-sulphur concentrations, indicating low risk of PAG, with the exception of the saprolite. The saprolite and transition zone samples contained very low NP, whereas the waste rock and low-grade economic material had NP related to reactive carbonate minerals. The saprolite samples were classified primarily as acid generating and PAG, whereas the other waste rock and low-grade economic material samples were classified as NPAG. This is not expected to change based on the inclusion of the Sona Hill Deposit.

The tailings samples contained low to negligible sulphide-sulphur concentrations and were classified as NPAG. The majority of the NP of the tailings was associated with the reactive carbonate minerals and/or lime added during the metallurgical testing. The saprolite tailings contained little to no reactive carbonate minerals, and thus the NP present in the saprolite tailings was related to the lime added during the metallurgical process.

Leachate testing indicated that the waste rock may develop alkaline drainage with the possibility of elevated concentrations of aluminum, selenium, chromium and, to a lesser extent, Cu, and phosphorus.

The tailings could develop alkaline drainage with the possibility of elevated concentrations of aluminum, selenium, chromium, arsenic, cobalt, copper, iron, molybdenum, WAD cyanide and sulphate. The TSF design assumes that the natural low permeability of the surficial soils, and the lower concentrations of elements in the TSF pond due to attenuation from natural degradation, settling, and mixing with precipitation, which averages about 2.6 m annually, will reduce concentrations in any TSF discharge effluent to the aquatic receiving environment. Additional analysis (i.e., predictive water quality modelling) will be needed in a later phase to verify this assumption.

An EIA was prepared and submitted to the GGMC and Guyana EPA, which subsequently issued an environmental permit for mining and processing. ETK submitted an amendment to its Environmental Management Plan in October of 2021 to include the processing of silver from the deposits and adding the Southeast Area of the Main Toroparu Deposit and the Sona Hill Deposit to the permitted operations under the Environmental Authorization. EPA accepted the revised Environmental Plan on November 22, 2021.

The final mining permit will be required prior to commencing full-scale operations.

25.11 Capex and Opex Costs

LoM capital requirement is estimated at US$1,299 million. This estimate is broken down into the following items:

- Initial capital is estimated at US$355 million, of which around US$41 million is pre-stripping costs and US$314 million directly related to the installation of the Project facilities;
- The mineral processing infrastructure is programmed to be expanded in year 6 to include a flotation circuit, the capital cost of this expansion is estimated at US$281 million (includes mine fleet costs and other items outside of process infrastructure); and
- Sustaining capital over the LoM is estimated at US$662 million.

Based on the assumptions presented in this report, only the initial capital will require financing, while the expansion capital should be financed by the Project's free cash flow.

Financing requirement is estimated at US$217 million, as it is assumed that a metal stream deal with Wheaton will fund around US$138 million of the initial capital.

25.12 Economic Analysis

Project evaluation resulting economics present an after-tax net present value of US$794 million, at 5% discount rate, and an internal rate of return of 46.08%.

These economic results include the precious metal purchase agreement with Wheaton, with installment payments (PMPA Installments) estimated at US$138 million of the initial capital.

Economic results indicate a LoM AISC cash cost of US$916/Au-oz. Sensitivity analysis indicate that the Project is most sensitive to variations of metal prices followed by operating costs.

25.13 Risks and Uncertainties

There are some risks that are inherent to a mining project. The PEA is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would enable them to be categorized as Mineral Reserves. There is no certainty that the PEA will result in an operating mine. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Areas of uncertainty that may materially impact the PEA and Mineral Resource Estimate include:

- Changes to long term metal price assumptions.

- Changes to the input values for mining, processing, and G&A costs to constrain the estimate.
- Changes to local interpretations of mineralization geometry and continuity of mineralized zones.
- Changes to the density values applied to the mineralized zones.
- Changes to metallurgical recovery assumptions.
- Changes in assumptions of marketability of the final product.
- Variations in geotechnical, hydrogeological, and mining assumptions.
- Changes to assumptions with an existing agreement or new agreements.
- Changes to environmental, permitting, and social licence assumptions.
- EA Timing, requirements and supporting documentation.
- Logistics of securing and moving adequate services, labour and supplies could be affected by epidemics, pandemics, and other public health crises, including COVID-19 or similar such viruses.

25.14 Conclusions

The results of the PEA affirm the Project's technical and financial merits using base case and sensitivity metal price assumptions and the inputs in some areas from advanced historical studies completed by the Company that were at PFS or FS levels.

The Company believes there is further potential to significantly expand the Mineral Resource and the mineral resources within the PEA mine plan. Under the assumptions presented in this Technical Report, and based on the available data, the Mineral Resources meet 2014 CIM Definition Standards, the 2019 CIM Best Practice Guidelines and show reasonable prospects of eventual economic extraction.

26 RECOMMENDATIONS

26.1 Introduction

The recommendations are focused on the completion of a PFS technical report that is predicated on additional infill drilling to increase the confidence of the resource, carry out further metallurgical test work, and various mine planning, processing related trade-off studies.

Table 26-1 tabulates the PFS recommendations which are anticipated to require a budget of US$7,826,500.

Table 26-1: Recommended PFS Budget

Item	Units	Unit Cost	Cost (US$)
10,000-metre infill drilling expansion drilling (used for metallurgy, geotechnical, etc.)	15,000	$200	$3,000,000
Underground Reserve development			$200,000
TSF and WMS Geotechnical Investigations			$240,000
Metallurgical and Comminution Testwork			$575,000
Mine Design Updates			$600,000
PFS Level Economic Assessment & Technical Report 43-101 (PFS)			$600,000
General support and administration costs, legal fees, professional fees, staff, fixed costs, etc.		40%	$1,900,000
Contingency (10%)		10%	$711,500
Total			**$7,826,500**

26.1.1 Mineral Resources and Mining Methods

The following recommendations are for mineral resources, open pit and underground mining mine plans:

- Complete infill drilling focusing on areas within the first five years of mining and to support metallurgical and geotechnical testwork.
- Validate development and stope support requirements via further geotechnical analysis.
- Conduct detailed mineable stope inventory, development design and sequencing analysis.
- Undertake testing of representative cemented paste backfill using various cement percentages to determine strength at various time periods.
- Conduct further geomechanical studies to improve the understanding of the rock mass characteristics and refine the underground mine design input.
- Conduct a geomechanical analysis of the interaction between the underground mine and open pit.
- Update and validate stope sizing restrictions via further geomechanical analysis.

- Optimize surface handling.
- Conduct pit portal location study.
- Conduct infill drilling between the main and northwest pits.
- Conduct hydrogeological studies to obtain better understanding of underground dewatering requirements.
- Monthly mine plans for the first three to four years and quarterly after the monthly plans.
- Detailed stockpile and waste dump progression designs.
- Further optimization of smaller starter pits to minimize stripping and stockpiling.
- Additional trade-off studies to when underground and float plant should come online.
- Conduct further geomechanical studies to improve the understanding of the rock mass characteristics and refine the underground mine design input.
- Conduct geotechnical drilling and analysis for the Sona Hill Deposit.
- Conduct a feasibility level geomechanical analysis for the Sona Hill area.
- Update and validate stope sizing restrictions via further geomechanical analysis.
- Optimize stockpile handling.

26.1.2 Mineral Processing and Metallurgical Testing

The following recommendations are made with regard to metallurgical testing:

- Elevated temperature leach tests on Sona Hill variability (high Te) samples.
- Testwork to determine relationship between gold recovery, tellurium grade and tellurium deportment across the Sona Hill Deposit.
    - Expanded variability testing to better understand cyanide soluble copper characterization, leach and flotation behaviour based on revised feed blends and sources.
    - Expanded variability testing to collate additional data for metallurgical performance predictions.
    - Perform carbon characterization tests under variable cyanide soluble copper concentrations.
    - SAG mill parameter testing/determination and in the interim, circuit modelling based on currently known parameters.
    - Establish if high pH slurry streams will require acid dosing to effect cyanide detoxification system control.

A number of these test work proposals are being addressed by a test work program currently underway at Base Metal Laboratories, Kamloops, Canada which is anticipated to be completed in Q1, 2022.

Process design should ensure the materials handling aspects associated with saprolite feed including viscosities is adequately catered for.

It is estimated that the costs of this test work will be of the order of US$500,000. This excludes the costs of engineering oversight and the costs of procurement of the samples themselves.

26.1.3 Recovery Methods

Nordmin recommends the completion of the current feasibility level metallurgical study to finalize the process flowsheet and incorporate the test data into the design criteria and mass balance. This will require additional work and will rely on inputs from an updated ore processing (production) schedule.

Nordmin recommends monthly resolution open pit mine schedules or the first two years of the mine life and quarterly for the next three years for optimizing the design of the gold plant. The higher resolution schedule will further establish how the Sona Hill ores will be processed, either by blending, campaigning, or a mix of both.

During the prefeasibility study, trade-off evaluations, detailed engineering on the plant, equipment sizing, and cost estimation are recommended. Such evaluation being gold plant focused. Evaluations include:

- SAG mill amenability.
- Thickener locations for the CIL – feed thickening or tails thickening options exist which could benefit operating cost.
- Use of AARL elution in place of the originally proposed Zadra process.
- Carbon circuit modelling regarding number of tanks and configuration thereof. Use of package CIP contactors being an option to be explored.
- Revenue available as a function of increased CIL residence time. Sona Hill ores benefit from increased leach times. Incremental benefit may be obtained on other ore types, especially given the current metal prices.
- Consideration regarding tie-in of a future Flotation Plant and if any allowance is to be provided in the Gold Plant for services in the future.
- Water treatment and water management of the TSF supernatant if necessary.

The option of SAG milling needs to be verified as a viable option for comminution. Should SAG milling be considered impractical or non-preferred the project would have to revert to dual feed circuits for saprolite and fresh (hardrock) ores.

The design of the carbon, elution and electrowinning circuits will need to consider slow carbon kinetics, low equilibrium loadings, elevated pH regimes, viscosity influences and cyanide soluble copper. The design engineer will need to ensure the specifics of the project are well catered for to avoid the CIL from under-performing.

The site wide water balance will need to be integrated with the plant water balance including forecasting out to include the first three years of flotation processing. This should be presented on a monthly basis to capture the two wet seasons experienced at site and ensure the TSF supernatant can be managed and to allow a more robust estimate of raw water demands for the process plant.

Compliance aspects with the ICMC will need to be defined so that the design engineer can align the detailed design accordingly.

Similarly, permitted discharge limits and/or TSF management constraints/levels will need to be provided to the engineer to ensure the process design is appropriate.

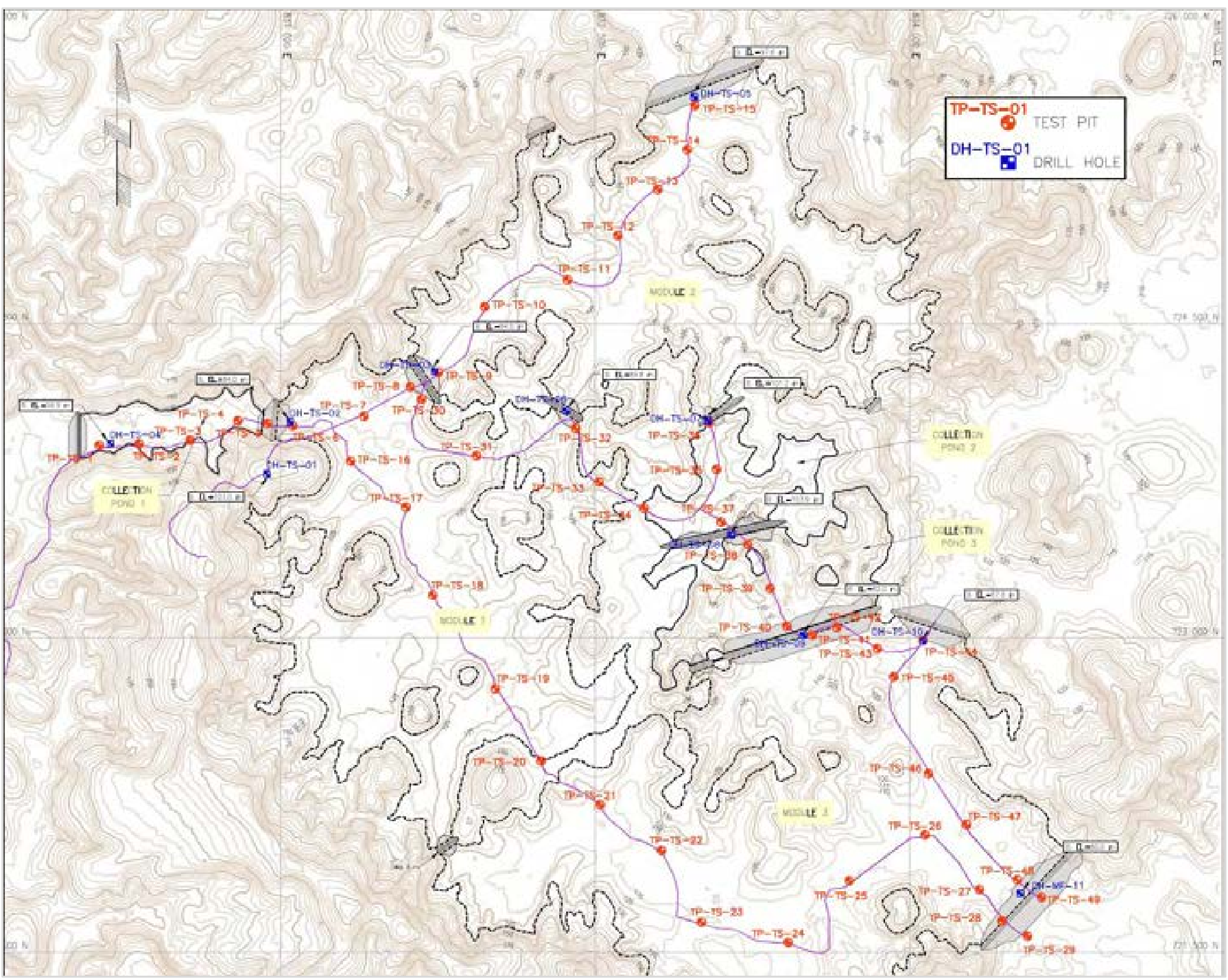
The estimated cost to complete this work is approximately US$575,000.

26.1.4 Project Infrastructure

26.1.4.1 Tailings Storage Facility

Additional geotechnical investigation within the foundation area of the dams, particularly in the central part and abutments. Figure 26-1 shows the geotechnical investigations developed and reported in the *Toroparu Bankable Feasibility Study Tailings Storage Facility Geotechnical Data Report* (KCB 2014). In general, the recommendations include the following:

- Carry out geotechnical investigations which include test pits, drill holes, SPT's (standard penetration tests), CPTu's (cone penetration tests with porewater pressure measurements), vane shear tests, and pocket penetrometer testing.
- Perform compaction field trials and develop a detailed instrumentation program for monitoring and interpretation of field performance.
- Develop the alignments/surface of the perimeter accesses in more detail to have more accurate quantities.
- Incorporate from the results of field investigations into final designs.

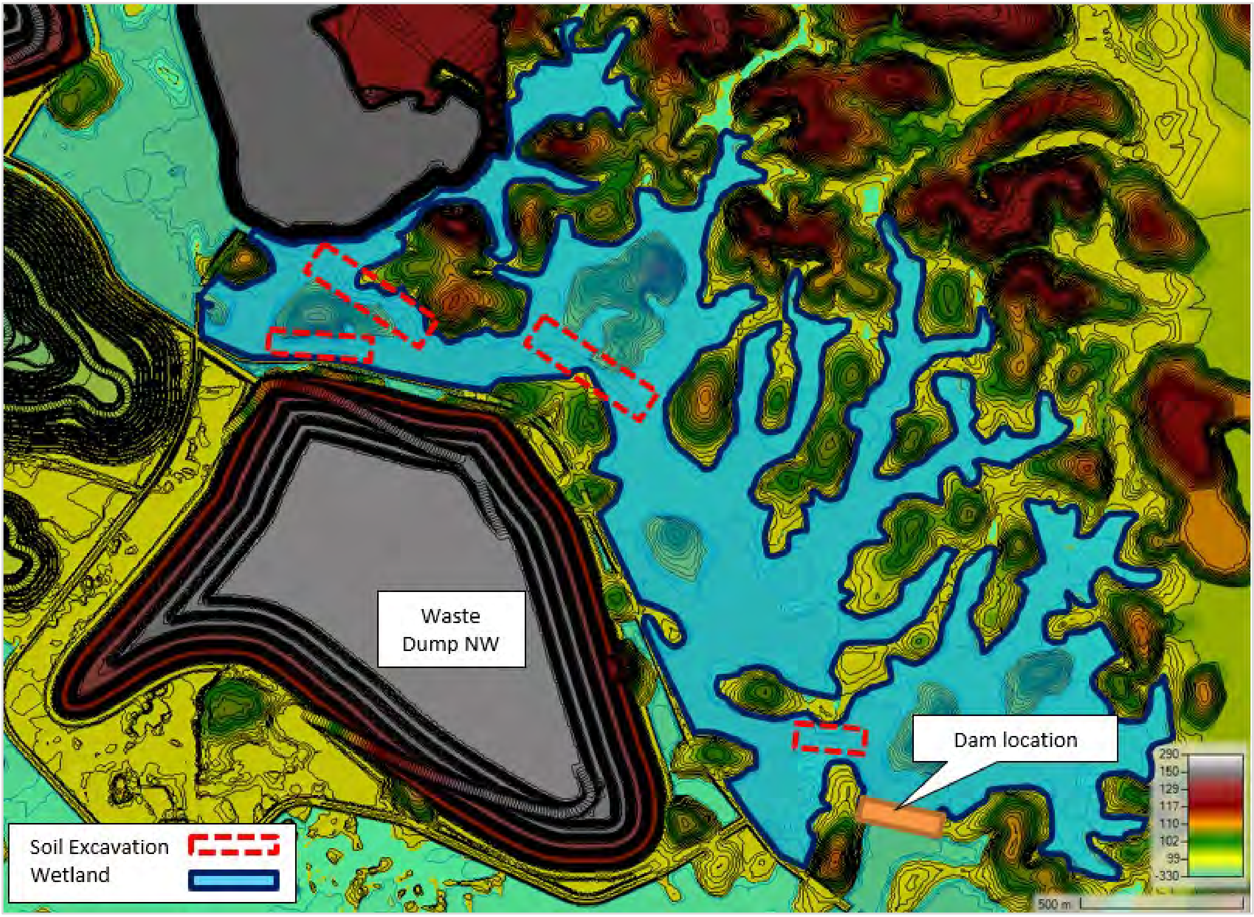


Source: KCB, 2021

Figure 26-1: Existing geotechnical investigations

26.1.4.2 WMS

- Excavate ditches at specific locations to allow the streamflow and discharge in the Wetland Area (Figure 26-2).



Source: KCB, 2021

Figure 26-2: WMS area ditch excavations

- Develop a water management system based on culverts in levees to release water trapped between mining facilities (dumps, waste rocks, pits, and haul roads) and channels to increase velocity of Wynamu River in low lying areas.
- Utilize rip rap protection to control slope erosion where water velocities approach erosion velocities (0.6 m/s – 3 m/s).
- Design levee protection from the Puruni River for the Northwest Pit during the mine life.
- Increase the top elevation of levee haul roads 98.5 m to 99 m to control inundation from floodwaters during heavy rainfall events.
- Integrate hydraulic infrastructure designs into master site plans for evaluation of potential interactions and continue hydrological information updates.
- Check the current haul road elevation design for a critical event;
- Develop water management maintenance plan to manage sediment transport issues.
- Review time retention and sedimentation and integrate with water quality studies.

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28 GLOSSARY

The Mineral Resources and Mineral Reserves have been classified according to CIM (CIM, 2014). Accordingly, the resources have been classified as Measured, Indicated, or Inferred, the reserves have been classified as proven, and probable based on the Measured and Indicated Resources as defined below.

28.1 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.

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 **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 Project. Geological evidence is derived from the 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.

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 Project. Geological evidence is derived from the 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.

28.2 Mineral Reserve

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 prefeasibility or feasibility level as appropriate that include the 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 prefeasibility study or feasibility study.

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.

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.

28.3 Definition of Terms

The following terms may be used in this Technical Report.

Table 28-1: Definition of Terms

Term	Definition
Assay	The chemical analysis of mineral samples to determine the metal content.
Capital Expenditure	All other expenditures not classified as operating costs.
Composite	Combining more than one sample result to give an average result over a larger distance.
Concentrate	A metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from the waste material in the ore.
Crushing	The initial process of reducing the ore particle size to render it more amenable for further processing.
Cut-Off Grade	The grade of mineralized rock, which determines as to whether or not it is economical to recover its gold content by further concentration.
Dilution	Waste, which is unavoidably mined with ore.
Dip	The angle of inclination of a geological feature/rock from the horizontal.
Fault	The surface of a fracture along which movement has occurred.
Footwall	The underlying side of an orebody or stope.
Gangue	Non valuable components of the ore.
Grade	The measure of the concentration of gold within the mineralized rock.
Hanging wall	The overlying side of an orebody or slope.
Haulage	A horizontal underground excavation which is used to transport mined ore.
Hydrocyclone	A process whereby material is graded according to size by exploiting centrifugal forces of particulate materials.
Igneous	Primary crystalline rock formed by the solidification of magma.
Kriging	An interpolation method of assigning values from samples to blocks that minimize the estimation error.

Term	Definition
Level	A horizontal tunnel, the primary purpose is the transportation of personnel and materials.
Lithological	Geological description pertaining to different rock types.
LRP	Long Range Plan.
Material Properties	Mine properties.
Milling	A general term used to describe the process in which the ore is crushed and ground and subjected to physical or chemical treatment to extract the valuable metals to a concentrate or finished product.
Mineral/Mining Lease	A lease area for which mineral rights are held.
Mining Assets	The Material Properties and Significant Exploration Properties.
Ongoing Capital	Capital estimate of a routine nature, which is necessary for sustaining operations.
Ore reserve	See Mineral Reserve.
Pillar	Rock left behind to help support the excavations in an underground mine.
Sedimentary	Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks.
Shaft	An opening cut downwards from the surface for transporting personnel, equipment, supplies, ore, and waste.
Sill	A thin, tabular, horizontal to the sub horizontal body of igneous rock formed by the injection of magma into planar zones of weakness.
Smelting	A high-temperature pyrometallurgical operation conducted in a furnace, in which the valuable metal is collected to a molten matte or dolt phase and separated from the gangue components that accumulate in a less dense molten slag phase.
Stope	The underground void created by mining.
Stratigraphy	The study of stratified rocks in terms of time and space.
Strike	The direction of the line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction.
Sulphide	A sulphur-bearing mineral.
Tailings	Finely ground waste rock from which valuable minerals or metals have been extracted.
Thickening	The process of concentrating solid particles in suspension.
Total Expenditure	All expenditures, including those of an operating and capital nature.
Variogram	A statistical representation of the characteristics (usually grade).

28.4 Abbreviations, Acronyms, and Symbols

The following abbreviations, acronyms, and symbols may be used in this Technical Report.

Table 28-2: Abbreviations, Acronyms, and Symbols

Abbreviation/Acronym/Symbol	Term
%	percent
<	less than
>	greater than
°	degree (degrees)
°C	degrees Celsius
µm	micrometre or micron
AA	atomic absorption
AARL	Anglo American Research Laboratories
AAS	Atomic absorption spectrometry
ACO	Average Copper Ore
ADR	Adsorption, Desorption and Recovery
ADT	Articulated dump trucks
AG	Acid Generating
Ag	silver
AISC	all-in sustaining cost
AP	Acid Potential
ARD	acid rock drainage
Au	gold
BAT	best available techniques
BBWI	Bond Ball Work Index
BG	Background Grade
BHEM	Borehole Electromagnetic
BIF	banded-iron formation
BMA	Bulk Mineral Analyses
BML	Base Metallurgical Laboratories
BMP	best management practices
BOO	built, owned, and operated
Capex	capital expenditure
CCME	Canadian Council of Ministers of the Environment
CD	Contact Water ditches
CGSZ	Central Guiana Shear Zone
CIF	Cost, insurance, and freight
CIL	carbon in leach
CIM	Canadian Institute of Mining, Metallurgy, and Petroleum
CIP	carbon in pulp

Abbreviation/Acronym/Symbol	Term
CITES	Convention on International Trade in Endangered Species of Wild Fauna and Flora
cm	centimetre
CMC	Carboxy Methyl Cellulose
CND	Cyanide destruction
CNP	determination of cyanide
CPB	Cemented paste backfill
CRM	certified reference material
DDH	diamond drill hole
DGPS	differential global positioning system
EIA	Environmental Impact Assessment
ELOS	Equivalent Linear Overbreak/Slough
EM	electromagnetic
EMPA	electron microprobe analysis
EPA	Environmental Protection Agency
EPC	Engineering, procurement and construction
ESMS	Environmental and Social Management System
ETZ	Equatorial Trough Zone
FAR	Fresh air raises
FDC	Fuel Distribution Company
FoS	Factor of Safety
FOT	Free on Truck
ft	foot (feet)
ft^2	square foot (feet)
ft^3	cubic foot (feet)
g	gram
G&A	General & Administrative
g/cm^3	grams per cubic centimetre
g/L	gram per litre
g/t	grams per tonne
Ga	giga-annum (1 billion years)
gal	gallon
GCM Mining or the Company	GCM Mining Corp.
GEMS	GEOVIA GEMS™
GFC	Guyana Forestry Commission
GGMC	Guyana Geology and Mining Commission
g-mol	gram-mole
gpm	gallons per minute
GPS	global positioning system
ha	hectare (10,000 m^2)

Abbreviation/Acronym/Symbol	Term
HCT	humidity cell testing
HMC	heavy mineral concentrate
HPGR	high pressure grinding roll
HSE	Health Safety and Environmental
HW	Hanging wall
ICMC	International Cyanide Management Code
ICP	induced couple plasma
ICP-AES	inductively coupled plasma atomic emission spectrometry
ID2	inverse distance squared
ID3	inverse distance cubed
IFC	International Finance Corporation
IFO	intermediate fuel oil
IP	induced polarization
IPP	independent power producer
IRR	internal rate of return
ISR	inductive source resistivity
ITCZ	Inter Tropical Convergence Zone
ITH	In-the-hole
IUCN	International Union for Conservation of Nature
KCB	Klohn Crippen Berger
kg	kilogram
km	kilometre
km^2	square kilometre
KRHP	Kumarau River Hydroelectric Project
kt	thousand tonnes
KV	kriging variance
L	litre
lb	pound
LCO	Low Copper Ore
LCT	Locked cycle test
LG	Low-Grade
LHOS	Longhole open stoping
LIDAR	light detection and ranging
LoM	Life of mine
m	metre
M	million
MA	Mechanical availability
Ma	mega annum (1 million years)
mbgs	metres below ground surface

Abbreviation/Acronym/Symbol	Term
MCC	motor control centres
MG	Medium Grade
mg/L	milligrams/litre
MgO	magnesium oxide
MIBC	methyl isobutyl carbinol
ML	Metal leaching
mm	millimetre
mm^2	square millimetre
mm^3	cubic millimetre
MMR	magnetometric resistivity
MOU	Memorandum of Understanding
Moz	million troy ounces
MP	Mining Permits
MSHA	Mine Safety and Health Administration
MSO	Mineable Shape Optimizer
Mt	million tonnes
MT	magnetotelluric
Mtpa	million tonnes per annum
NAG	net acid generation
NCD	non-contact water ditches
NI 43-101	Canadian National Instrument 43-101
NN	Nearest neighbour
NNW	north-northwest
NP	Neutralization Potential
NPAG	not-potentially acid generating
NPR	Net Potential Ratio
NS	north-south
OK	ordinary kriging
Opex	operating expenditures
oz	troy ounce
PAX	Potassium Amyl Xanthate
Pb	lead
PCOC	Potential Constituent of Concern
PEA	Preliminary Economic Assessment
PFS	Prefeasibility Study
PIMS	protocol independent multicasts
PL	Prospecting Licenses
PMA	particle mineral analysis
PMF	probable maximum flood
PMPA	Precious metal purchase agreement

Abbreviation/Acronym/Symbol	Term
PoF	Probability of Failure
POI	point of interconnection
PPA	power purchase agreement
ppb	parts per billion
ppm	parts per million
PPMS	Prospecting Permits Medium Scale
PSA	Pressure swing adsorption
QA	quality assurance
QC	quality control
QP	Qualified Persons
RAR	return air raises
RC	reverse circulation
RF	revenue factor
RMR	Rock Mass Rating
RoM	run of mine
RQD	rock quality designation
S	sulphur
SAG	semi-autogenous grinding
SC	spatial composites
SEC	Securities and Exchange Commission
SEDAR	System for Electronic Document Analysis and Retrieval
SFE	Shake Flask Extraction
SG	specific gravity
SI	Saprolite Intrusives
SI	saprolite intrusives
SIMS	secondary ion mass spectrometer
SMC	SAG mill comminution
SSE	south-southeast
SV	saprolite volcanics
t	tonne (metric ton) (2,204.6 pounds)
t/h	tonnes per hour
the Project	Toroparu Gold Project
the Property	Toroparu Gold Project Property
TML	Transportable moisture limit
TMS	trace mineral search
Toroparu Pit	Toroparu and NW Pits
tpd	tonnes per day
TSF	tailings storage facility
UC	Uncertain
UCS	Unconfined Compressive Strength

Abbreviation/Acronym/Symbol	Term
US	United States
UTM	Universal Transverse Mercator
VAT	Value Added Tax
VG	visible gold
VOIP	voice over internet protocol
VS	volcano-sedimentary
VSA	Vacation, sickness and absence
WAD	weak acid dissociable
WHO	World Health Organization
WMS	water management structures

Appendix A: QP Certificates of Authors

NORDMIN
RESOURCE & INDUSTRIAL
ENGINEERING

CERTIFICATE OF QUALIFIED PERSON

I, Kurt Boyko, P. Eng., of Thunder Bay, Ontario do hereby certify:

1. I am the Consulting Specialist – Mechanical Systems with Nordmin Engineering Ltd. with a business address at 160 Logan Ave., Thunder Bay, Ontario.
2. This certificate applies to the Amended Technical Report titled "Revised NI 43-101 Technical Report and Preliminary Economic Assessment for the Toroparu Gold Project, Upper Puruni River Region of Western Guyana" with an Effective Date of December 1, 2021 (the "Amended Technical Report").
3. I am a graduate of Lakehead University, 1994, with a Bachelor of Engineering Degree, Mechanical.
4. I am a member in good standing of the Professional Engineers of Ontario and registered as a Professional Engineer, license number 90418484.
5. My relevant experience includes 29 years of experience of design and operation of industrial processing plants, including materials handling and pumping design, machine design, mine dewatering plans, and ventilation systems. I am a "Qualified Person" for the purposes of Canadian National Instrument 43-101 ("Revised NI 43-101" or the "Instrument").
6. I have not inspected the site of the Toroparu Gold Project (the "Project"), located in Upper Puruni River Region of Western Guyana.
7. I am responsible for Section 13 and 17 and portions of Sections 1, 18, 21, 25 and 26 (the "Relevant Sections") summarized within the Amended Technical Report.
8. I am independent of GCM Mining Corp., as defined in Section 1.5 of the Instrument.
9. I have read the NI 43-101 reporting requirements and the Relevant Sections of the Amended Technical Report, for which I am responsible, have been prepared in compliance with the Instrument and Form 43-101F1.
10. As of the date of the Amended Technical Report, to the best of my knowledge, information, and belief, the Relevant Sections of the Technical Report that I am responsible for, contain all scientific and technical information relating to the Project that is required to be disclosed to make the Amended Technical Report not misleading.
11. I have no prior involvement with the Project that is the subject of the Amended Technical Report.

Signed and dated this 4th day of February 2022, at Thunder Bay, Ontario.

LICENSED PROFESSIONAL ENGINEER
K.A. BOYKO
90418484
04-FEB-22
PROVINCE OF ONTARIO

____

Kurt Boyko, P.Eng.
Consulting Specialist – Mechanical Systems
Nordmin Engineering Ltd.

NORDMIN
RESOURCE & INDUSTRIAL
ENGINEERING

CERTIFICATE OF QUALIFIED PERSON

I, Glen Kuntz, P. Geo., of Thunder Bay, Ontario do hereby certify:

1. I am the Consulting Specialist – Geology/Mining formerly with Nordmin Engineering Ltd. with a business address at 160 Logan Ave., Thunder Bay, Ontario, Canada, P7A 6R1.
2. This certificate applies to the technical report titled " Revised NI 43-101 Technical Report and Preliminary Economic Assessment for the Toroparu Gold Project, Upper Puruni River Region of Western Guyana" with an Effective Date of December 1, 2021 (the "Amended Technical Report").
3. I am a graduate of the University of Manitoba, 1991, with a Bachelor of Science in Geology.
4. I am a member in good standing of the Association of Professional Geoscientist of Ontario and registered as a Professional Geoscientist, license number 0475.
5. My relevant experience includes 30 years of experience in exploration, operations and resource estimations. I am a "Qualified Person" for the purposes of National Instrument 43-101 ("NI 43-101" or the "Instrument").
6. My most recent personal inspection of the Toroparu Gold Project (the "Project"), located in Upper Puruni River Region of Western Guyana, was April 24 and April 25, 2021. The Project is within the Mazaruni Mining District.
7. I am responsible for Sections 2 through 12 (excepting 5.5.2, 5.5.4), 14, 19, 20, 23, 24 and 27 and portions of Sections 1, 25 and 26 (the "Relevant Sections") within the Amended Technical Report.
8. I am independent of GCM Mining Corp., as defined in Section 1.5 of the Instrument.
9. I have read the NI 43-101 reporting requirements and the Relevant Sections of the Technical Report, for which I am responsible, have been prepared in compliance with the Instrument and Form 43-101F1.
10. As of the date of the Amended Technical Report, to the best of my knowledge, information, and belief, the Relevant Sections of the Technical Report that I am responsible for, contain all scientific and technical information relating to the Project that is required to be disclosed to make the Technical Report not misleading.
11. I have no prior involvement with the Project that is the subject of the Technical Report.

Signed and dated this 4th day of February 2022, at Thunder Bay, Ontario.

Glen Kuntz, P.Geo.
Consulting Specialist – Geology/Mining (former)
Nordmin Engineering Ltd.

PROFESSIONAL GEOSCIENTIST
Feb 4/22
GLEN N. KUNTZ
PRACTISING MEMBER
0475
ONTARIO

CERTIFICATE OF QUALIFIED PERSON

I, Ben Peacock, P. Eng., of North Bay, Ontario do hereby certify:

1. I am a Senior Engineer with Knight Piésold Ltd. with a business address at 1650 Main Street West, North Bay, Ontario, Canada, P1B 8G5.
2. This certificate applies to the Amended Technical Report titled “Revised NI 43-101 Amended Technical Report and Preliminary Economic Assessment for the Toroparu Gold Project, Upper Puruni River Region of Western Guyana” with an Effective Date of December 1, 2021 (the “Amended Technical Report”).
3. I am a graduate of the University of Waterloo, 2008 with a Bachelor of Applied Science in Civil Engineering.
4. I am a member in good standing of the Professional Engineers of Ontario and registered as a Professional Engineer, license number 100141409.
5. My relevant experience includes 13 years of experience as a consulting engineer in the field of mining rock mechanics. I am a “Qualified Person” for the purposes of Canadian National Instrument 43-101 (“NI 43-101” or the “Instrument”).
6. I have not visited the Toroparu Gold Project situated in the Upper Puruni River Region of Western Guyana.
7. I am responsible for Section 16.4.2 and the related portions of Sections 1 and 26 (the “Relevant Sections”) within the Amended Technical Report.
8. I am independent of GCM Mining Corp., as defined in Section 1.5 of the Instrument.
9. I have read the NI 43-101 reporting requirements and the Relevant Sections of the Amended Technical Report for which I am responsible have been prepared in compliance with the Instrument and Form 43-101F1.
10. As of the date of the Amended Technical Report, to the best of my knowledge, information, and belief, the Relevant Sections of the Amended Technical Report that I am responsible for, contain all scientific and technical information relating to the Toroparu Gold Project that is required to be disclosed to make the Amended Technical Report not misleading.
11. I have no prior involvement with the Toroparu Gold Project that is the subject of the Amended Technical Report.

Signed and dated this 4th day of February 2022, at North Bay, Ontario.

LICENSED PROFESSIONAL ENGINEER
B. D. PEACOCK
100141409
2022-02-04
PROVINCE OF ONTARIO

Ben Peacock, P.Eng
Senior Engineer
Knight Piésold Ltd.

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www.srk.com

CERTIFICATE OF QUALIFIED PERSON

I, Fernando Rodrigues, BS Mining, MBA, MMSAQP do hereby certify that:

1. I am Practice Leader and Principal Consultant (Mining Engineer) of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the Technical Report titled “Revised NI 43-101 Technical Report and Preliminary Economic Assessment for the Toroparu Gold Project, Upper Puruni River Region of Western Guyana” with an Effective Date of December 1, 2021 (the “Amended Technical Report”).
3. I graduated with a Bachelors of Science degree in Mining Engineering from South Dakota School of Mines and Technology in 1999. I am a QP member of the MMSA. I have worked as a Mining Engineer for a total of 23 years since my graduation from South Dakota School of Mines and Technology in 1999. My relevant experience includes mine design and implementation, short term mine design, dump design, haulage studies, blast design, ore control, grade estimation, database management, reserve estimation and due diligences.
4. I have read the definition of “qualified person” set out in National Instrument 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 fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
5. I visited the Toroparu property on multiple occasions February 12, 2012, March 15, 2014 and June 23, 2017for a total of 7 days.
6. I am responsible for the open pit minable resource estimate, open pit mining section, open pit mining cost sections and the economic analysis Sections 16.1, 16.2, 16.3 and 22 and portions of Sections 1, 25, 26 and capex/opex costs Section 21 within the Amended Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Amended Technical Report. The nature of my prior involvement is to provide pit and waste dump design, pit optimization, mining cost estimation, mineral reserve and minable resource estimation.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Amended Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Amended Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Amended Technical Report not misleading.

Dated this 4th Day of February, 2022.

Signed ______________________ ***Stamped***

Fernando Rodrigues, BS Mining, MBA, MMSAQP [01405QP]

CERTIFICATE OF QUALIFIED PERSON

I, David Willms, P. Eng., of Nanaimo, British Columbia do hereby certify:

1. I am a Senior Geotechnical Engineer with Klohn Crippen Berger Ltd. with a business address at 500 – Virtual Way, Vancouver, British Columbia, Canada, V5M 4X6.
2. This certificate applies to the Amended Technical Report titled "Revised NI 43-101 Technical Report and Preliminary Economic Assessment for the Toroparu Gold Project, Upper Puruni River Region of Western Guyana" with an Effective Date of February 4, 2022 (the "Amended Technical Report").
3. I am a graduate of the University of British Columbia, 2003, with a Bachelor of Applied Science in Geological Engineering, and 2010 with an M.Eng. in Geological Engineering.
4. I am a member in good standing of Engineers and Geoscientists British Columbia and registered as a Professional Engineer, registration number 33062.
5. My relevant experience includes more than 18 years of experience in the mining industry, with a focus on tailings management. I am a "Qualified Person" for the purposes of National Instrument 43-101 ("NI 43-101" or the "Instrument").
6. I have not visited the Project site.
7. I am responsible for Sections 1.8.2, 1.8.3, 5.5.2, 5.5.4, 18.2, 18.3, 25.9, 26.1.4.1, and 26.1.4.2 (the "Relevant Sections") within the Amended Technical Report.
8. I am independent of GCM Mining Corp., as defined in Section 2.5 of the Instrument.
9. I have read the NI 43-101 reporting requirements and the Relevant Sections of the Amended Technical Report, for which I am responsible, have been prepared in compliance with the Instrument and Form 43-101F1.
10. As of the date of the Amended Technical Report, to the best of my knowledge, information, and belief, the Relevant Sections of the Amended Technical Report that I am responsible for, contain all scientific and technical information relating to the Project that is required to be disclosed to make the Amended Technical Report not misleading.
11. I have no prior involvement with the Project that is the subject of the Amended Technical Report.

Signed and dated this 4th day of February 2022, at Nanaimo, British Columbia.

____

David Willms, P.Eng.
Senior Geotechnical Engineer, Associate
Klohn Crippen Berger Ltd.

NORDMIN
RESOURCE & INDUSTRIAL
ENGINEERING

CERTIFICATE OF QUALIFIED PERSON

I, Brian Wissent, P. Eng., of Kitchener, Ontario do hereby certify:

1. I am the Senior Engineer with Nordmin Engineering Ltd. with a business address at 160 Logan Ave., Thunder Bay, Ontario, Canada, P7A 6R1.
2. This certificate applies to the Amended Technical Report titled “Revised NI 43-101 Technical Report and Preliminary Economic Assessment for the Toroparu Gold Project, Upper Puruni River Region of Western Guyana” with a Mineral Resource Effective Date of December 1st, 2021 (the “Amended Technical Report”).
3. I am a graduate of Dalhousie University, 2007 with a Bachelor of Mineral Resource Engineering.
4. I am a member in good standing of the Association of Professional Engineers of Ontario and registered as a Professional Engineer, license number 100193972.
5. My relevant experience includes 15 years of experience as a mining engineer in mining companies (Barrick, Goldcorp) and in consulting firms (Nordmin, Global Mine Design), with specific expertise in underground planning, geotechnical and designs. I am a “Qualified Person” for the purposes of National Instrument 43-101 (“NI 43-101” or the “Instrument”).
6. I have not visited the Toroparu Gold Project (the “Project”), located in the Upper Puruni River Region of Western Guyana. The Project is within the Mazaruni Mining District.
7. I am responsible for the underground portions of Section 16.1, 16.2, Section 16.4.1 and Sections 16.4.3 through 16.4.19, and their related portion of Sections 1, 21, 25, 26 (the “Relevant Sections”) within the Amended Technical Report.
8. I am independent of GCM Mining Corp., as defined in Section 1.5 of the Instrument.
9. I have read the NI 43-101 reporting requirements and the Relevant Sections of the Amended Technical Report, for which I am responsible, have been prepared in compliance with the Instrument and Form 43-101F1.
10. As of the date of the Amended Technical Report, to the best of my knowledge, information, and belief, the Relevant Sections of the Amended Technical Report that I am responsible for, contain all scientific and technical information relating to the Project that is required to be disclosed to make the Amended Technical Report not misleading.
11. I have no prior involvement with the Project that is the subject of the Amended Technical Report.

Signed and dated this 4th day of February 2022, at Kitchener, Ontario.

LICENSED PROFESSIONAL ENGINEER
B. J. WISSENT
100193972
2022/02/04
PROVINCE OF ONTARIO

Brian Wissent, P.Eng.
Senior Engineer
Nordmin Engineering Ltd.

CERTIFICATE OF QUALIFIED PERSON

I, Daniel Y. Yang, M.Eng., P. Eng., of Vancouver, British Columbia do hereby certify:

1. I am a Specialist Geotechnical Engineer with Knight Piésold Ltd. with a business address at Suite 1400, 750 West Pender Street, Vancouver, British Columbia, Canada V6C 2T8.
2. This certificate applies to the Amended Technical Report titled "Revised NI 43-101 Technical Report and Preliminary Economic Assessment for the Toroparu Gold Project, Upper Puruni River Region of Western Guyana" with an Effective Date of December 1, 2021 (the "Amended Technical Report").
3. I graduated with a Bachelor degree of Civil Engineering from Tongji University in China in 1992 and obtained a Master degree of Geotechnical Engineering from the University of Alberta in Canada in 2002.
4. I am a member in good standing of the Engineers and Geoscientists British Columbia and registered as a Professional Engineer, license number 28936.
5. My relevant experience includes 28 years of experience in geotechnical site investigation, slope stability assessment, open pit slope design, and pit dewatering planning for various mining projects. I am a "Qualified Person" for the purposes of Canadian National Instrument 43-101 ("NI 43-101" or the "Instrument").
6. I visited the Toroparu Gold Project (the "Project"), located in Upper Puruni River Region of Western Guyana, on August 4 to 6, 2010 and February 11 to 14, 2014. The Project is within the Mazaruni Mining District.
7. I am responsible for Sections 16.3.2.1, 16.3.2.2, and the related portions of Sections 1 and 26.
8. I am independent of GCM Mining Corp., as defined in Section 1.5 of the Instrument.
9. I have read the NI 43-101 reporting requirements and the portions of the Amended Technical Report for which I am responsible have been prepared in compliance with the Instrument and Form 43-101F1.
10. As of the date of the Amended Technical Report, to the best of my knowledge, information, and belief, the Sections of the Amended Technical Report that I am responsible for, contain all scientific and technical information relating to the Toroparu Gold Project that is required to be disclosed to make the Amended Technical Report not misleading.

Signed and dated this 4th day of February 2022, at Vancouver, British Columbia.

PROFESSIONAL
PROVINCE OF
Y. YANG
28936
BRITISH COLUMBIA
ENGINEER
2022-02-04

Daniel Y. Yang, P.Eng,
Specialist Geotechnical Engineer
Knight Piésold Ltd.

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Appendix B: Properties List

PROSPECTING PERMITS

GS8Number	PPMS Number	Area (Acres)	Location	Map Number	Renewal Date	Comments
A-106/014/500/95	164/2000	846	Puruni River	25NW	6-Jun	JOINT VENTURE WITH ALPHONSO
A-106/015/501/95	165/2000	1,051	Puruni River	25NW	6-Jun	JOINT VENTURE WITH ALPHONSO
A-106/016/502/95	166/2000	936	Puruni River	25NW	6-Jun	JOINT VENTURE WITH ALPHONSO
A-106/017/503/95	167/2000	962	Puruni River	24NE/25NW	6-Jun	Part of Mining Licence; owned by ETK
A-106/018/504/95	168/2000	953	Puruni River	24NE/25NW	6-Jun	JOINT VENTURE WITH ALPHONSO
A-140/011/258/97	0090/2000	1,105	Upper Mazaruni	24NE	10-Feb	JOINT VENTURE WITH ALPHONSO
A-140/012/97	0467/2002	1,077	Upper Mazaruni	24NE	7-Jul	JOINT VENTURE WITH ALPHONSO
A-140/013/97	0659/2002	1,065	Upper Puruni	24NE	6-Oct	JOINT VENTURE WITH ALPHONSO
A-140/014/97	0660/2002	1,085	Upper Puruni	24NE	6-Oct	JOINT VENTURE WITH ALPHONSO
A-140/018/97	0663/2002	1,035	Tamakay	24NE	6-Oct	JOINT VENTURE WITH ALPHONSO
A-140/019/97	0664/2002	1,106	Tamakay	24NE	6-Oct	JOINT VENTURE WITH ALPHONSO
A-140/020/97	0665/2002	1,133	Upper Puruni	24NE	6-Oct	JOINT VENTURE WITH ALPHONSO
A-140/021/268/97	0523/2001	1,072	Tamakay	24NE/24SE	27-Aug	JOINT VENTURE WITH ALPHONSO
A140/023/270/97	0091/2000	1,058	Puruni River	24NE/24SE	10-Feb	JOINT VENTURE WITH ALPHONSO
A-140/024/271/97	0092/2000	1,176	Puruni River	24SE	10-Feb	JOINT VENTURE WITH ALPHONSO
A-140/025/272/95	0093/2000	1,126	Puruni River	24SE	10-Feb	JOINT VENTURE WITH ALPHONSO
A-140/026/273/97	0094/2000	957	Puruni River	24SE	10-Feb	JOINT VENTURE WITH ALPHONSO
A-140/027/274/97	0095/2000	828	Puruni River	24SE	10-Feb	JOINT VENTURE WITH ALPHONSO
A-140/028/0275/97	0195/2001	1,200	Puruni River	24SE	13-Mar	JOINT VENTURE WITH ALPHONSO
A-140/030/97	0667/2002	1,012	Tamakay	24NE/24SE	6-Oct	JOINT VENTURE WITH ALPHONSO
A-184/000/0394/99	0264/2001	848	Puruni River	24NE	11-Mar	JOINT VENTURE WITH ALPHONSO
A-184/002/0396/99	0266/2001	1134	Ikuk River	24NE	11-Mar	JOINT VENTURE WITH ALPHONSO
A-184/009/99	0579/2002	819	Upper Puruni	24NE	15-Aug	JOINT VENTURE WITH ALPHONSO
A-184/010/99	0580/2002	822	Upper Puruni	24NE	15-Aug	JOINT VENTURE WITH ALPHONSO
A-184/011/99	0581/2002	862	Upper Puruni	24NE	15-Aug	JOINT VENTURE WITH ALPHONSO
A-184/012/99	0582/2002	1087	Upper Puruni	24NE	15-Aug	JOINT VENTURE WITH ALPHONSO
A-184/013/99	0583/2002	1200	Upper Puruni	24NE	15-Aug	JOINT VENTURE WITH ALPHONSO
A-185/001/99	0577/2002	879	Upper Puruni	24NE	15-Aug	JOINT VENTURE WITH ALPHONSO
A-185/002/99	0578/2002	1,081	Upper Puruni	24NE	14-Aug	JOINT VENTURE WITH ALPHONSO
A-185/003/0411/99	0227/2001	1,009	Puruni River	24NE	7-Mar	JOINT VENTURE WITH ALPHONSO
A-185/007/0415/99	0330/2001	1,107	Upper Puruni	24NE	6-Mar	JOINT VENTURE WITH ALPHONSO
A-185/008/0416/99	0331/2001	1,067	Upper Puruni	24NE	6-Mar	JOINT VENTURE WITH ALPHONSO
A-185/009/0417/99	0424/2001	1,157	Upper Puruni	24NE	27-May	JOINT VENTURE WITH ALPHONSO
A-185/013/0421/99	0333/2001	1,078	Upper Puruni	24NE	7-Mar	JOINT VENTURE WITH ALPHONSO
A-185/014/0422/99	0334/2001	1,200	Upper Puruni	24NE	6-Mar	JOINT VENTURE WITH ALPHONSO
A-185/015/0423/99	0335/2001	1,200	Ikuk River	24NE	6-Mar	JOINT VENTURE WITH ALPHONSO
A-185/016/0424/99	0336/2001	649	Ikuk River	24NE	6-Mar	JOINT VENTURE WITH ALPHONSO
A-185/019/0427/99	0339/2001	622	Upper Puruni	24NE	6-Mar	JOINT VENTURE WITH ALPHONSO
A-185/020/0428/99	0340/2001	689	Upper Puruni	24NE	6-Mar	JOINT VENTURE WITH ALPHONSO
A-185/021/0429/99	0341/2001	660	Upper Puruni	24NE	6-Mar	JOINT VENTURE WITH ALPHONSO
A-185/024/0432/99	0344/2001	1,180	Ikuk River	24NE	7-Mar	JOINT VENTURE WITH ALPHONSO
A-185/025/0433/99	0345/2001	1,067	Ikuk River	24NE	8-Mar	JOINT VENTURE WITH ALPHONSO
A-185/026/0426/99	0346/2001	742	Putaring	24NE	8-Mar	JOINT VENTURE WITH ALPHONSO
A-185/027/99	0697/2002	868	Upper Puruni	24NE	16-Oct	JOINT VENTURE WITH ALPHONSO
A-185/028/0436/99	0347/2001	1,179	Putaring	24NE	7-Mar	JOINT VENTURE WITH ALPHONSO
A-185/029/0437/99	0348/2001	1,166	Putaring	24NE	7-Mar	JOINT VENTURE WITH ALPHONSO
A-185/030/0438/99	0349/2001	1,093	Putaring	24NE	8-Mar	JOINT VENTURE WITH ALPHONSO
A-185/031/0439/99	0350/2001	1,200	Putaring	24NE	8-Mar	JOINT VENTURE WITH ALPHONSO
A-185/032/0440/99	0351/2001	1,200	Putaring	24NE	6-Mar	JOINT VENTURE WITH ALPHONSO
A-185/033-0441/99	0352/2001	1,200	Putaring	24NE	6-Mar	JOINT VENTURE WITH ALPHONSO
A-185/035/0443/99	0354/2001	1,124	Puruni River	24NE	8-Mar	JOINT VENTURE WITH ALPHONSO
A-199/000/2000	620/2001	1,085	Puruni River	25NW	19-Sep	JOINT VENTURE WITH ALPHONSO
A-199/021/2000	639/2001	1,091	Puruni River	24NE	20-Sep	JOINT VENTURE WITH ALPHONSO
A-199/022/2000	640/2001	1,020	Puruni River	24NE	20-Sep	JOINT VENTURE WITH ALPHONSO
A-199/023/2000	641/2001	1,045	Puruni River	24NE	20-Sep	JOINT VENTURE WITH ALPHONSO
A-199/024/2000	642/2001	1,008	Puruni River	24NE	20-Sep	JOINT VENTURE WITH ALPHONSO
A-199/025/2000	643/2001	1,057	Puruni River	24NE	20-Sep	JOINT VENTURE WITH ALPHONSO
A-199/033/2000	0644/2002	1,047	Tamakay	24NE	7-Oct	JOINT VENTURE WITH ALPHONSO
A-199/035/2000	0646/2002	1,013	Tamakay	24NE	7-Oct	JOINT VENTURE WITH ALPHONSO
A-199/038/00	0649/2002	1,114	Upper Puruni	25NW	8-Oct	JOINT VENTURE WITH ALPHONSO
A-199/039/00	0686/2002	852	Upper Puruni	25NW	8-Oct	JOINT VENTURE WITH ALPHONSO
A-199/040/00	0687/2002	897	Upper Puruni	25NW	8-Oct	JOINT VENTURE WITH ALPHONSO
A-218/001/2001	0678/2002	421	Tamakay	24SE	15-Oct	JOINT VENTURE WITH ALPHONSO
A-302/001	0672/2003	389	Puruni River	24SE	5-Nov	JOINT VENTURE WITH ALPHONSO
A-302/002	0671/2003	556	Puruni River	24SE	5-Nov	JOINT VENTURE WITH ALPHONSO
Total Acres		**64,570**				

MINING PERMITS

GS8 Number	MP Number	Area (Acres)	Location	Map Number	Renewal Date	5th Year Renewal Date for Licence	Comments
A-4/MP/000/	007/2004	1123	Mazuruni	24NE	28-Apr	2024-04-28	JOINT VENTURE WITH ALPHONSO
A-4/MP/001/	008/2004	1117	Mazuruni	24NE	28-Apr	2024-04-28	JOINT VENTURE WITH ALPHONSO
A-4/MP/002/	009/2004	1200	Mazuruni	24NE	28-Apr	2024-04-28	Part of Mining Licence; owned by ETK
A-4/MP/003/	010/2004	1145	Mazuruni	24NE	28-Apr	2024-04-28	Part of Mining Licence; owned by ETK
A-4/MP/004/	011/2004	414	Mazuruni	24NE	28-Apr	2024-04-28	Part of Mining Licence; owned by ETK
A-4/MP/005/	012/2004	858	Mazuruni	24NE	28-Apr	2024-04-28	Part of Mining Licence; owned by ETK
A-4/MP/006/	013/2004	1098	Mazuruni	24NE	28-Apr	2024-04-28	JOINT VENTURE WITH ALPHONSO
A-4/MP/007/	014/2004	992	Mazuruni	24NE	28-Apr	2024-04-28	JOINT VENTURE WITH ALPHONSO
A-4/MP/008/	015/2004	1145	Mazuruni	24NE	28-Apr	2024-04-28	Part of Mining Licence; owned by ETK
A-4/MP/009/	016/2004	893	Mazuruni	24NE	28-Apr	2019-04-28	JOINT VENTURE WITH ALPHONSO
A-40/MP/000	253/2010	1158	Mazuruni	24SE	1-Nov	2020-11-29	JOINT VENTURE WITH ALPHONSO
A-89/MP/000	410/2013	1133	Mazuruni	24SE/24NE	1-Nov	2018-11-15	JOINT VENTURE WITH ALPHONSO
A-95/MP/000	313/2013	1000	Mazuruni	24NE	1-Sep	2023-09-03	JOINT VENTURE WITH ALPHONSO
A-194/MP/000	609/2014	1200	Mazuruni	24NE	1-Aug	2019-08-04	JOINT VENTURE WITH ALPHONSO
A-195/MP/000	610/2014	955	Mazuruni	24NE	1-Aug	2019-08-04	JOINT VENTURE WITH ALPHONSO
A-196/MP/000	611/2014	1119	Mazuruni	24NE	1-Aug	2019-08-04	JOINT VENTURE WITH ALPHONSO
A-182/MP/000	161/2014	291	Mazuruni	24NE	1-Jun	2019-06-20	JOINT VENTURE WITH ALPHONSO
A-111/MP/000	415/2013	1026	Mazuruni	24NE	1-Oct	2018-10-10	JOINT VENTURE WITH ALPHONSO
A-116/MP/000	419/2013	449	Mazuruni	24SE	1-Oct	2018-10-10	JOINT VENTURE WITH ALPHONSO
A-117/MP/000	420/2013	686	Mazuruni	24SE	1-Oct	2018-10-10	JOINT VENTURE WITH ALPHONSO
A-1022/MP/000	171/2017	1200	Mazuruni	24SE	27-Dec	2022-12-27	JOINT VENTURE WITH ALPHONSO
G-6/MP/000	007/2003	1190	Toroparu	24NE	9-Apr	2024-08-14	Part of Mining Licence; owned by ETK
G-6/MP/001	008/2003	1118	Toroparu	24NE	9-Apr	2024-08-14	Part of Mining Licence; owned by ETK
G-6/MP/002	009/2003	962	Toroparu	24NE	9-Apr	2024-08-14	Part of Mining Licence; owned by ETK
G-23/MP/000	278/2010	747	Toroparu	24NE	9-Apr	2015-12-20	Owned by ETK
Total Acres		**24,219.00**					

PROSPECTING LICENCES

GS8 Number	PL Number	Area (Acres)	Location	Map Number	Renewal Date	Comments
GS14:E-26	32/2013	9,570	Wynamu	16SE	21-Feb	Owned by ETK
GS14:E-27	33/2013	7,254	Wynamu East	16SE/17SW	21-Feb	Owned by ETK
Total Acres		**16,824.00**				

SMALL CLAIMS

HO NO	CLAIM NAME	Area (acres)	Location	Map Number	Renewal Date	HOLDER	Comments
31/1994/501	JOY #1	27	Toroparu	24NE	1-Mar	Alfro Alphonso	Part of Mining Licence; owned by ETK
31/1994/502	JOY #2	27	Toroparu	24NE	1-Mar	Alfro Alphonso	Part of Mining Licence; owned by ETK
31/1994/503	JOY #3	27	Toroparu	24NE	1-Mar	Alfro Alphonso	Part of Mining Licence; owned by ETK
31/1994/504	JOY #4	27	Toroparu	24NE	1-Mar	Alfro Alphonso	Part of Mining Licence; owned by ETK
31/1994/505	PAM #1	27	Toroparu	24NE	1-Mar	Alfro Alphonso	Part of Mining Licence; owned by ETK
31/1994/506	PAM #2	27	Toroparu	24NE	1-Mar	Alfro Alphonso	Part of Mining Licence; owned by ETK
31/1994/507	PAM #3	27	Toroparu	24NE	1-Mar	Alfro Alphonso	Part of Mining Licence; owned by ETK
Total Acres		**189.00**					

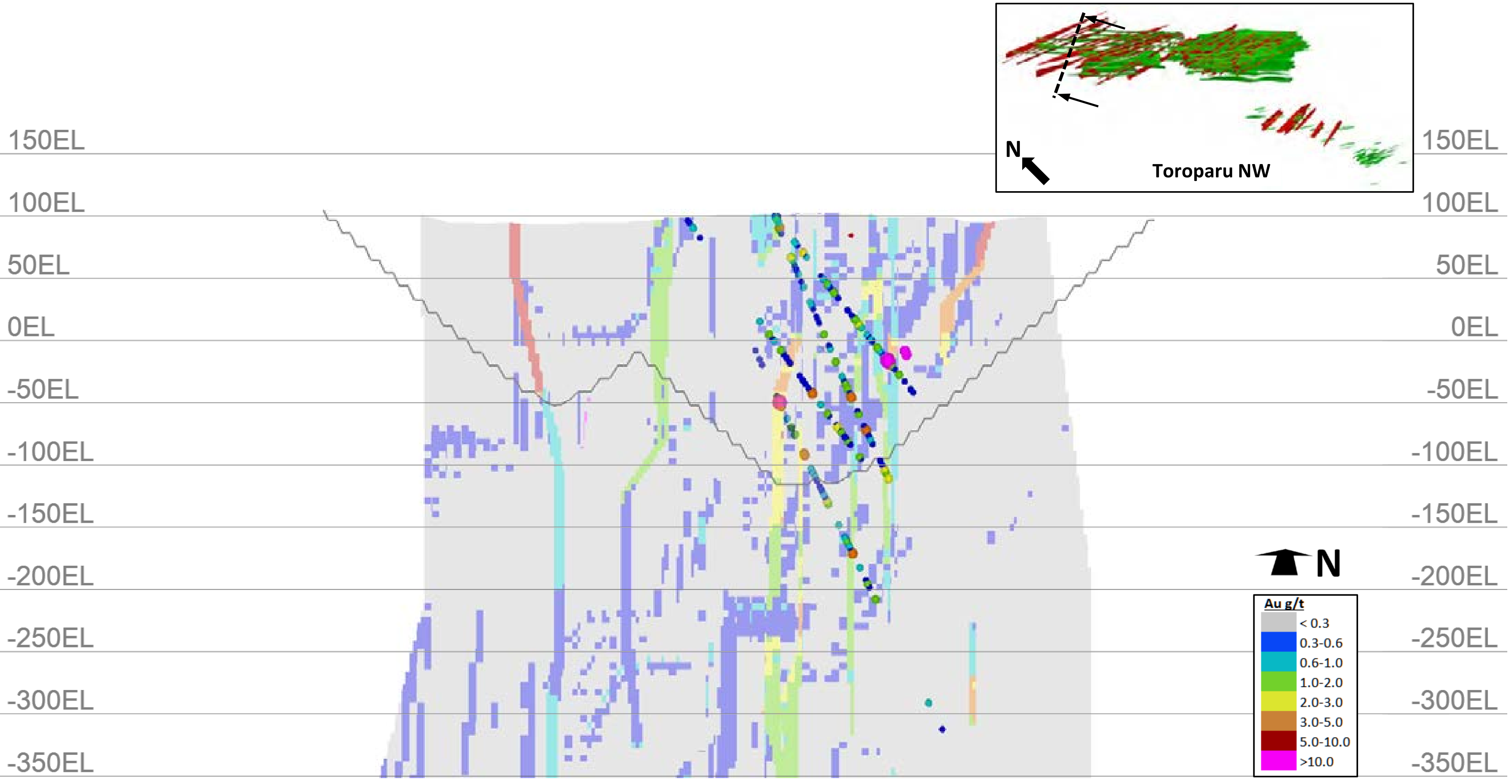
Overall Acreage **105,802**

GGMC Annual Rental Rates						
Anniversary Year	PL Rental ($ US/ac)	ML Rental ($ US/ac)	PPMS Rental ($ US/ac)	MP Rental ($ US/ac)	CP Rental $US/yr	CL Rental $US/yr
1	$0.50	$5.00	$0.25	$1.00	$2.51	$5.15
2	$0.60	$5.00	$0.35	$1.00	$2.51	$5.15
3	$1.00	$5.00	$0.45	$1.00	$2.51	$5.15
4	$1.50	$5.00	$0.55	$1.00	$2.51	$5.15
5	$2.00	$5.00	$0.65	$1.00	$2.51	$5.15
6	$3.00	$5.00	$0.75	$1.00	$2.51	$5.15
7	$3.00	$5.00	$0.85	$1.00	$2.51	$5.15
8	$3.00	$5.00	$0.95	$1.00	$2.51	$5.15
9	$3.00	$5.00	$1.05	$1.00	$2.51	$5.15
10	$3.00	$5.00	$1.15	$1.00	$2.51	$5.15
11	$3.00	$5.00	$1.25	$1.00	$2.51	$5.15
12	$3.00	$5.00	$1.35	$1.00	$2.51	$5.15
13	$3.00	$5.00	$1.45	$1.00	$2.51	$5.15
14	$3.00	$5.00	$1.55	$1.00	$2.51	$5.15
15	$3.00	$5.00	$1.65	$1.00	$2.51	$5.15
16	$3.00	$5.00	$1.75	$1.00	$2.51	$5.15
17	$3.00	$5.00	$1.85	$1.00	$2.51	$5.15
18	$3.00	$5.00	$1.95	$1.00	$2.51	$5.15
19	$3.00	$5.00	$2.05	$1.00	$2.51	$5.15
20	$3.00	$5.00	$2.15	$1.00	$2.51	$5.15

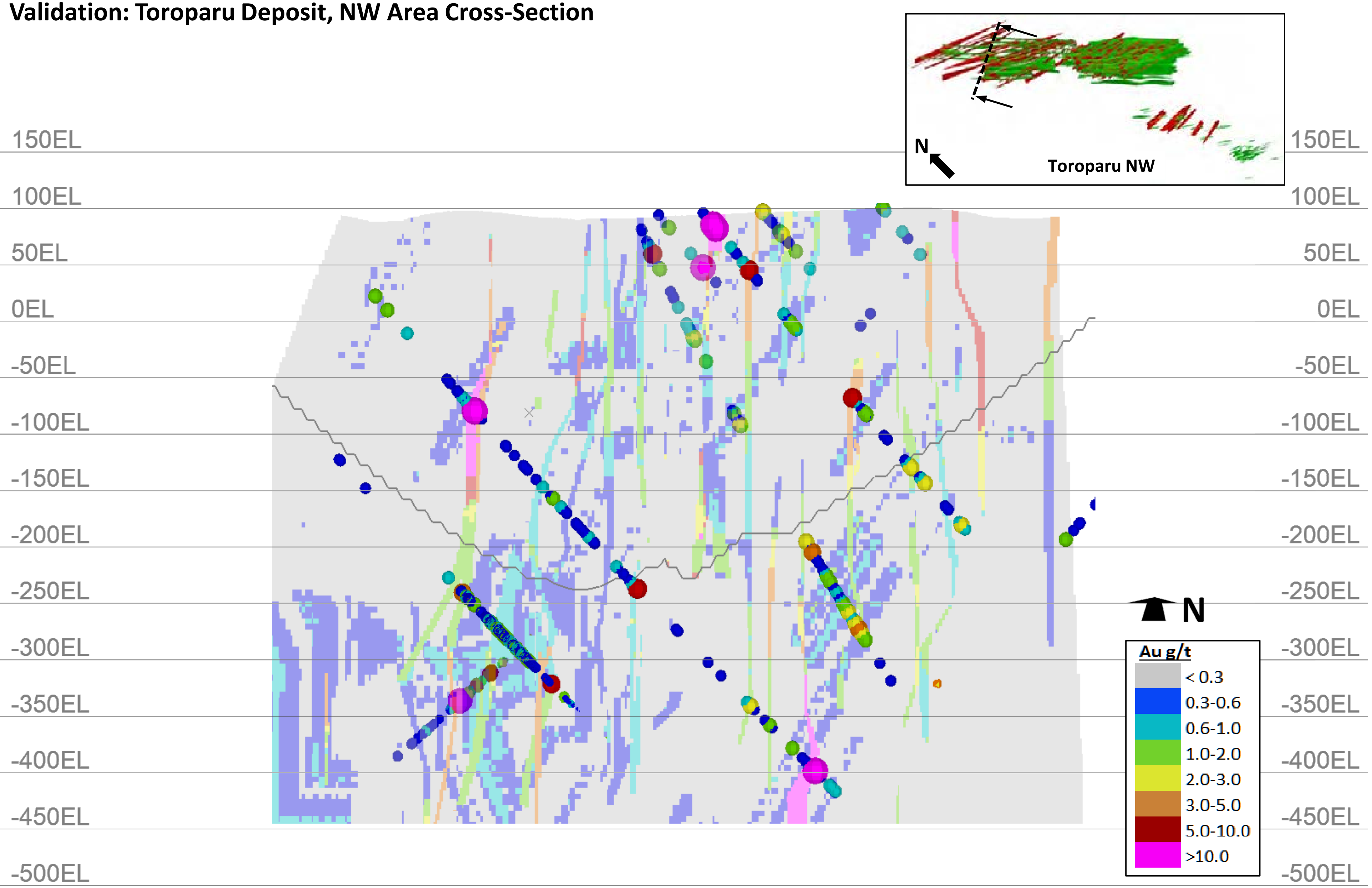
Claims are GY$2000 per year

Appendix C: Block Model Validation Images

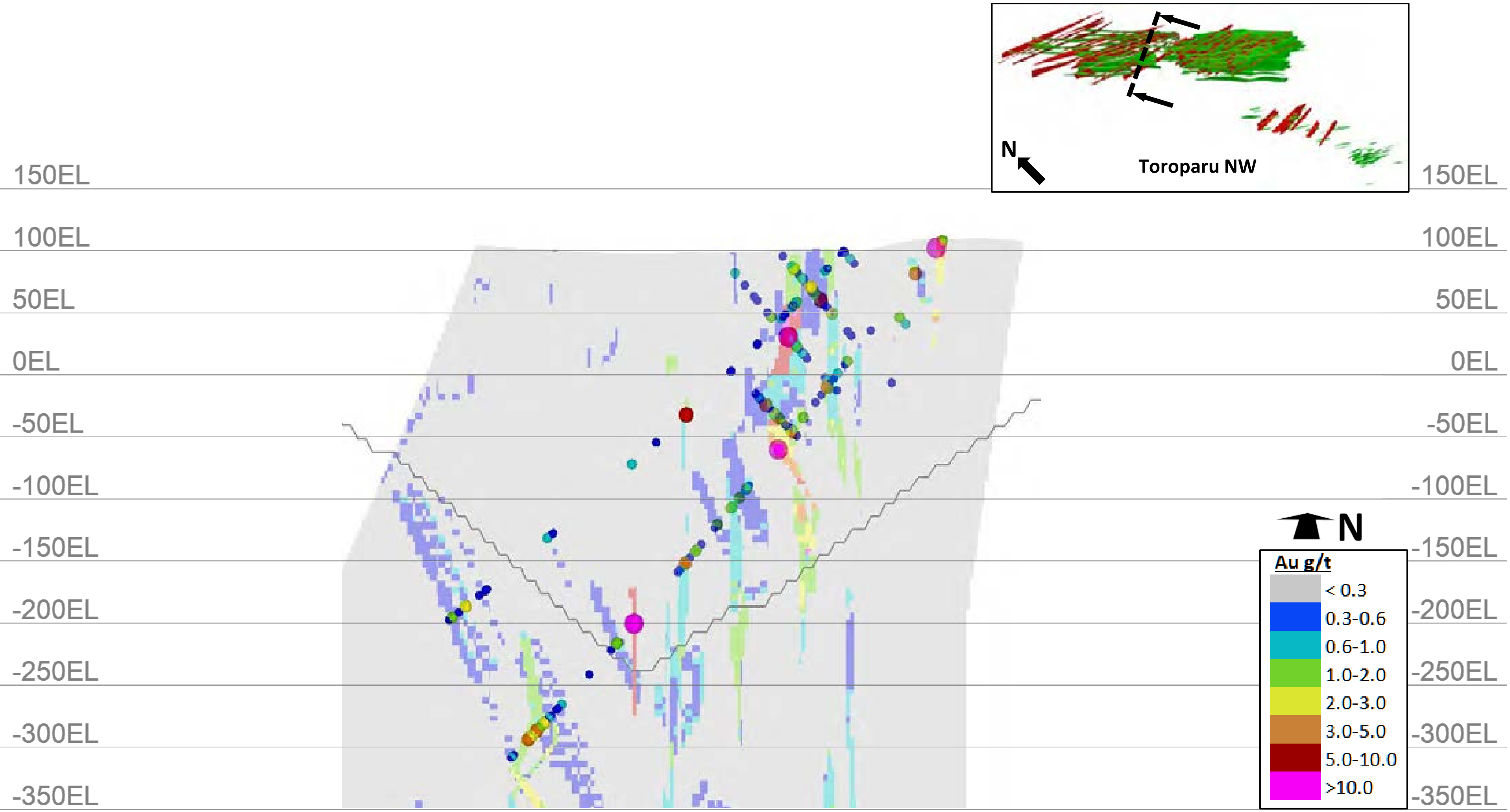
Validation: Toroparu Deposit, NW Area Cross-Section



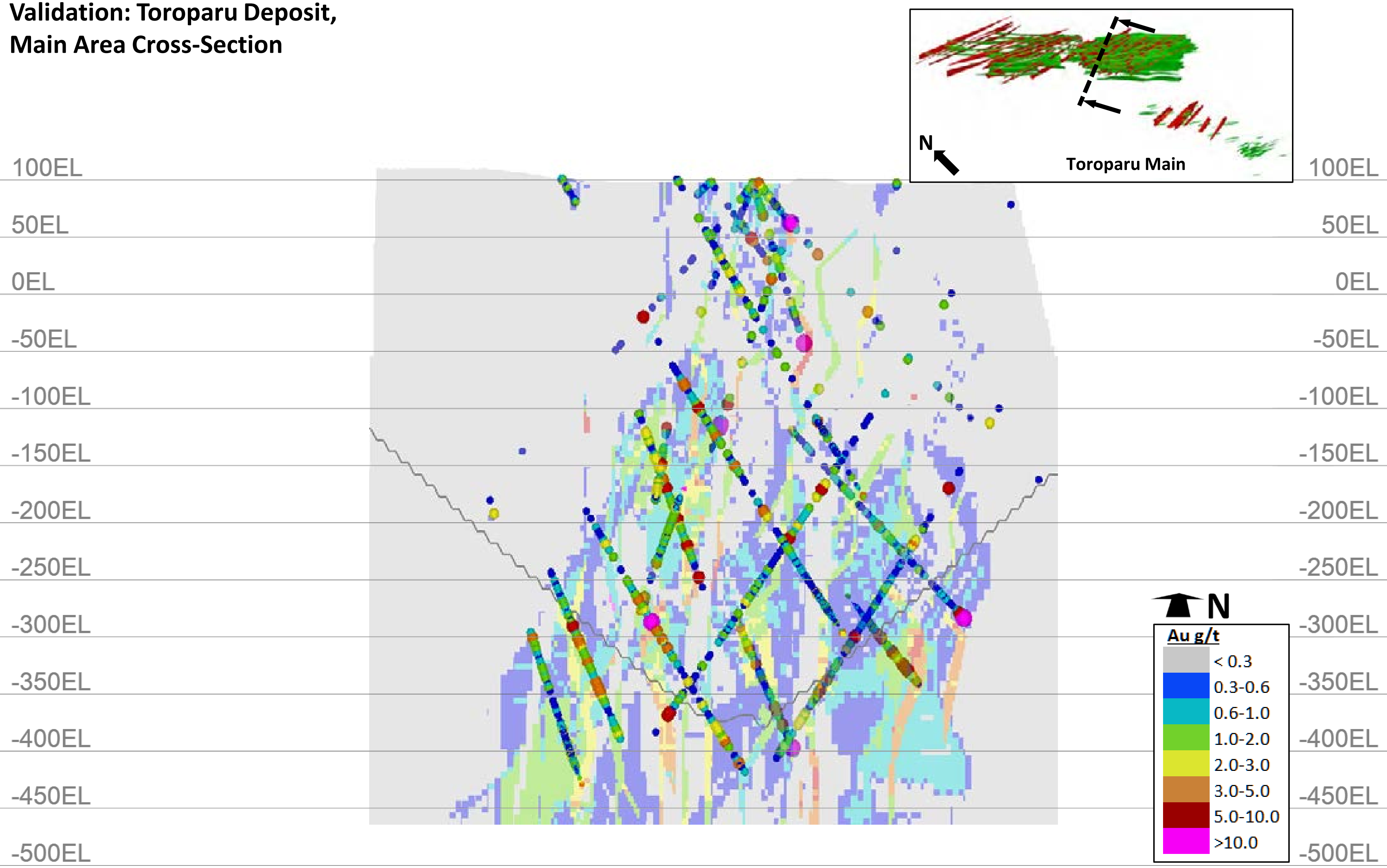
Validation: Toroparu Deposit, NW Area Cross-Section



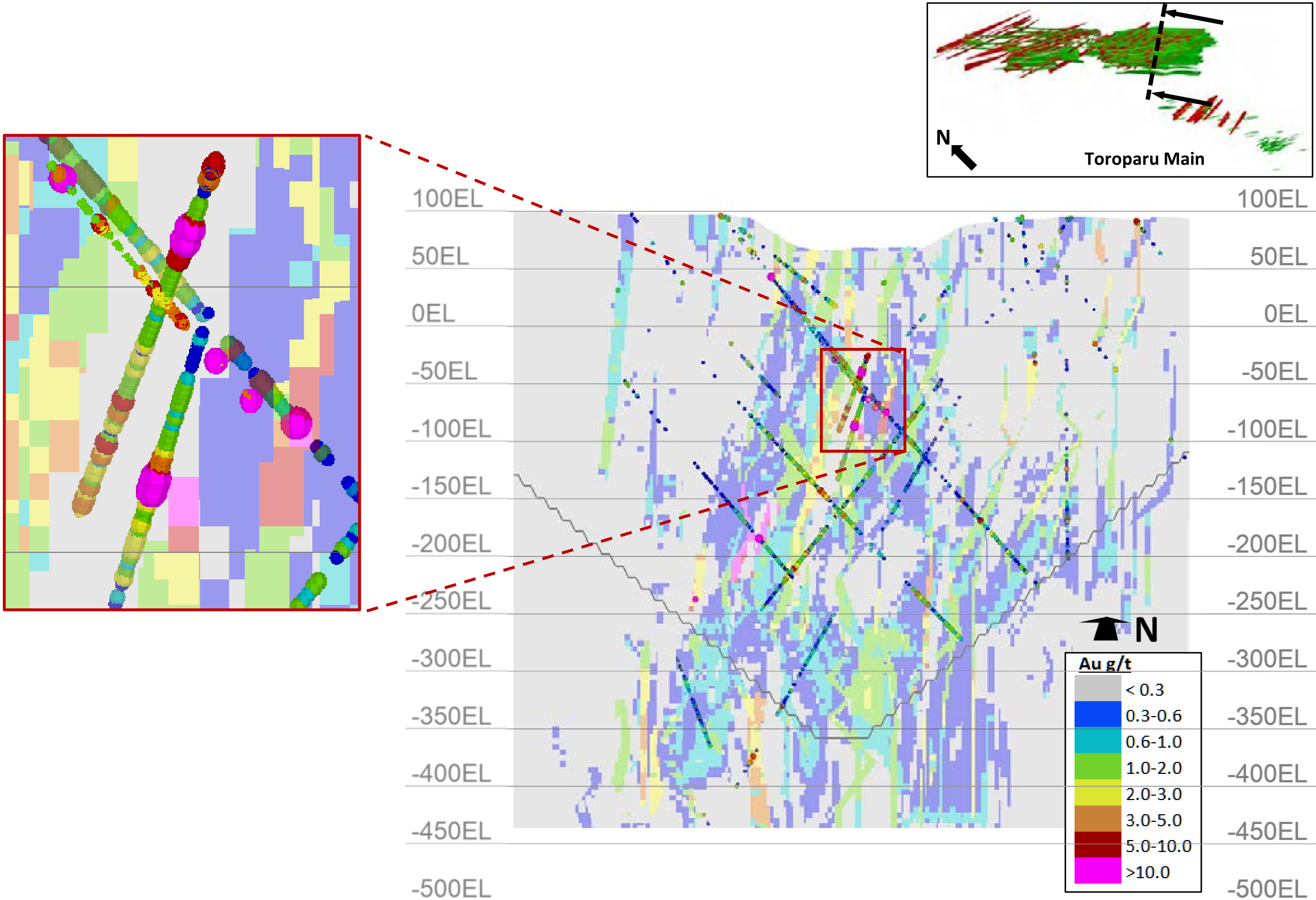
Validation: Toroparu Deposit, NW Area Cross-Section



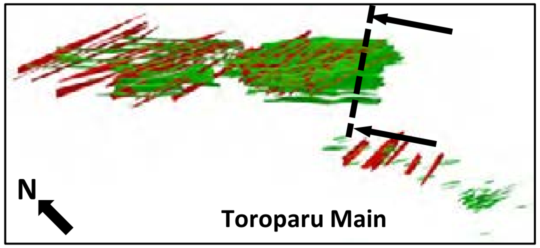
Validation: Toroparu Deposit, Main Area Cross-Section

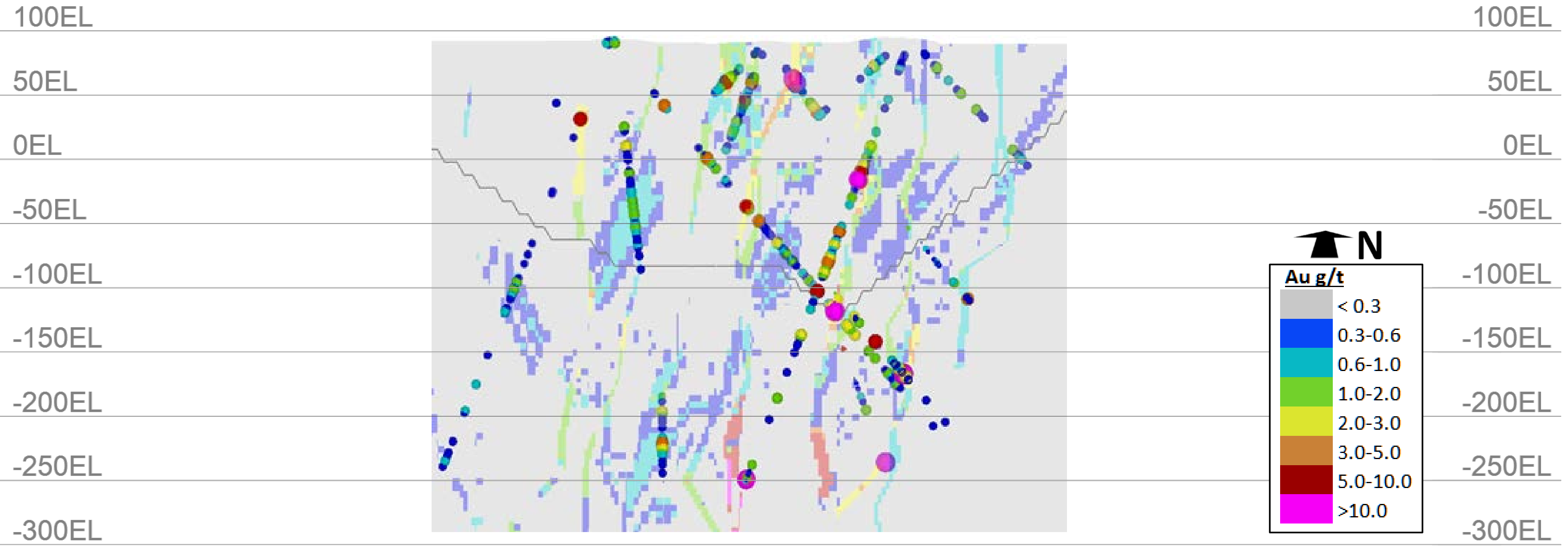


Validation: Toroparu Deposit, Main Area Cross-Section

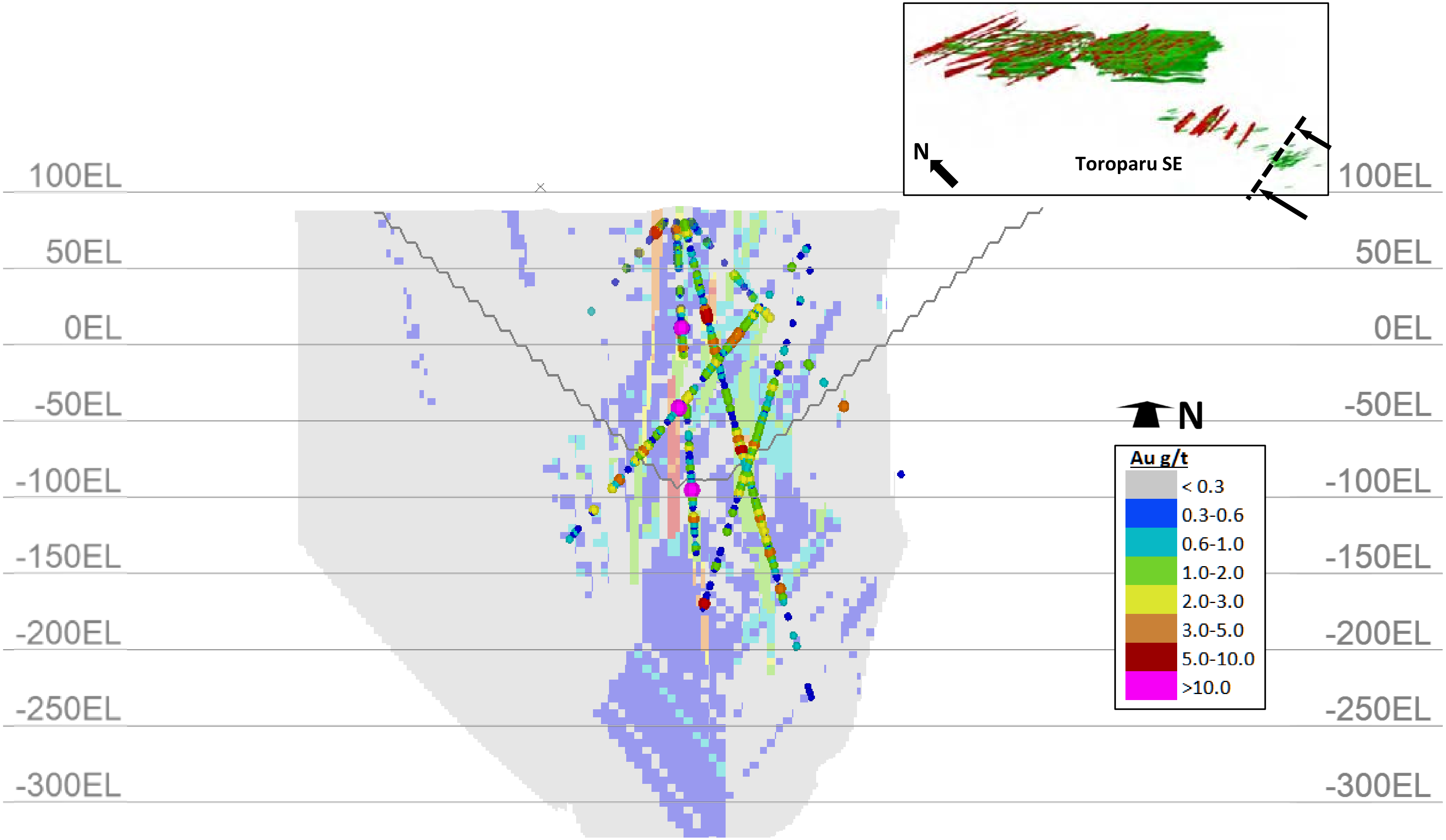


Validation: Toroparu Deposit, Main Area Cross-Section

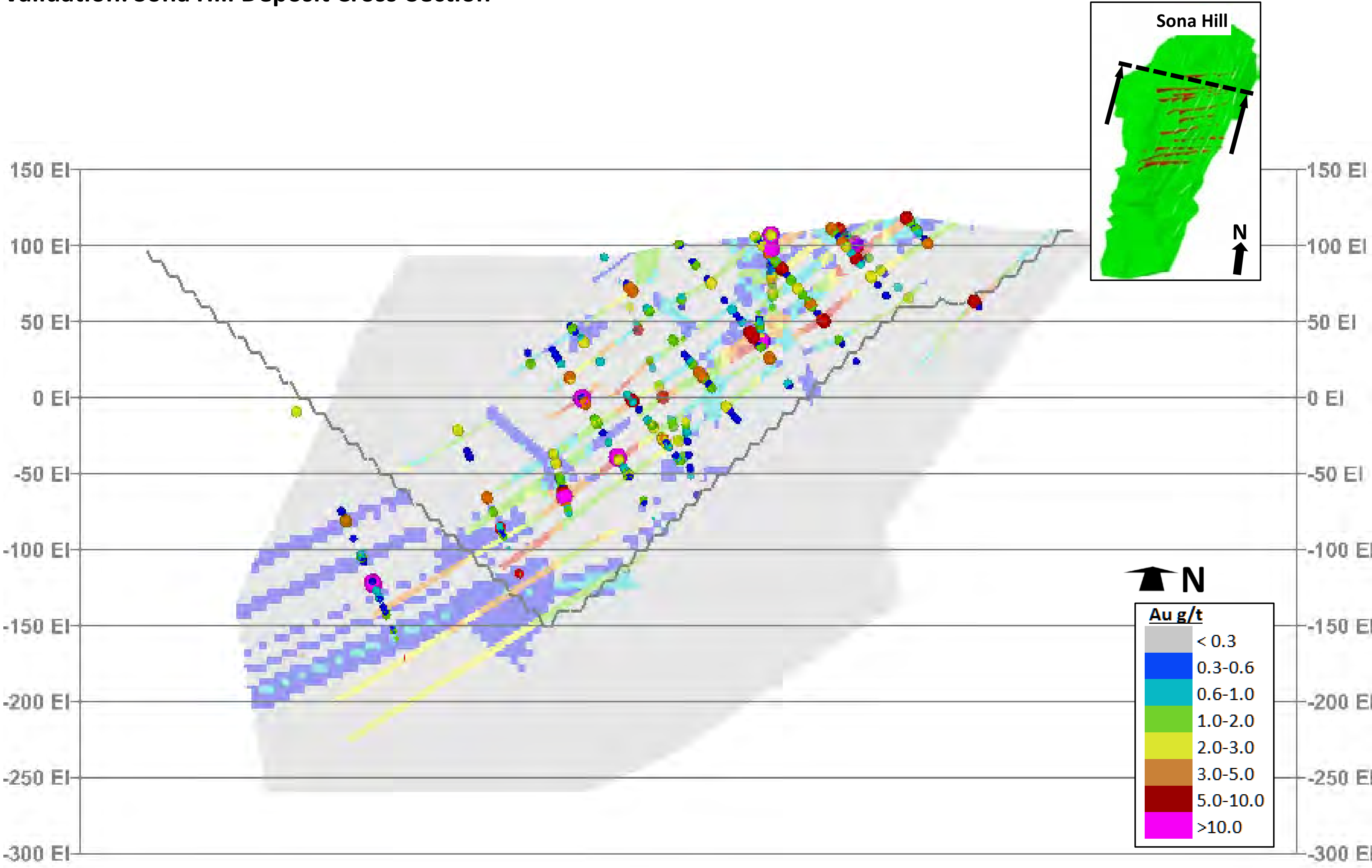




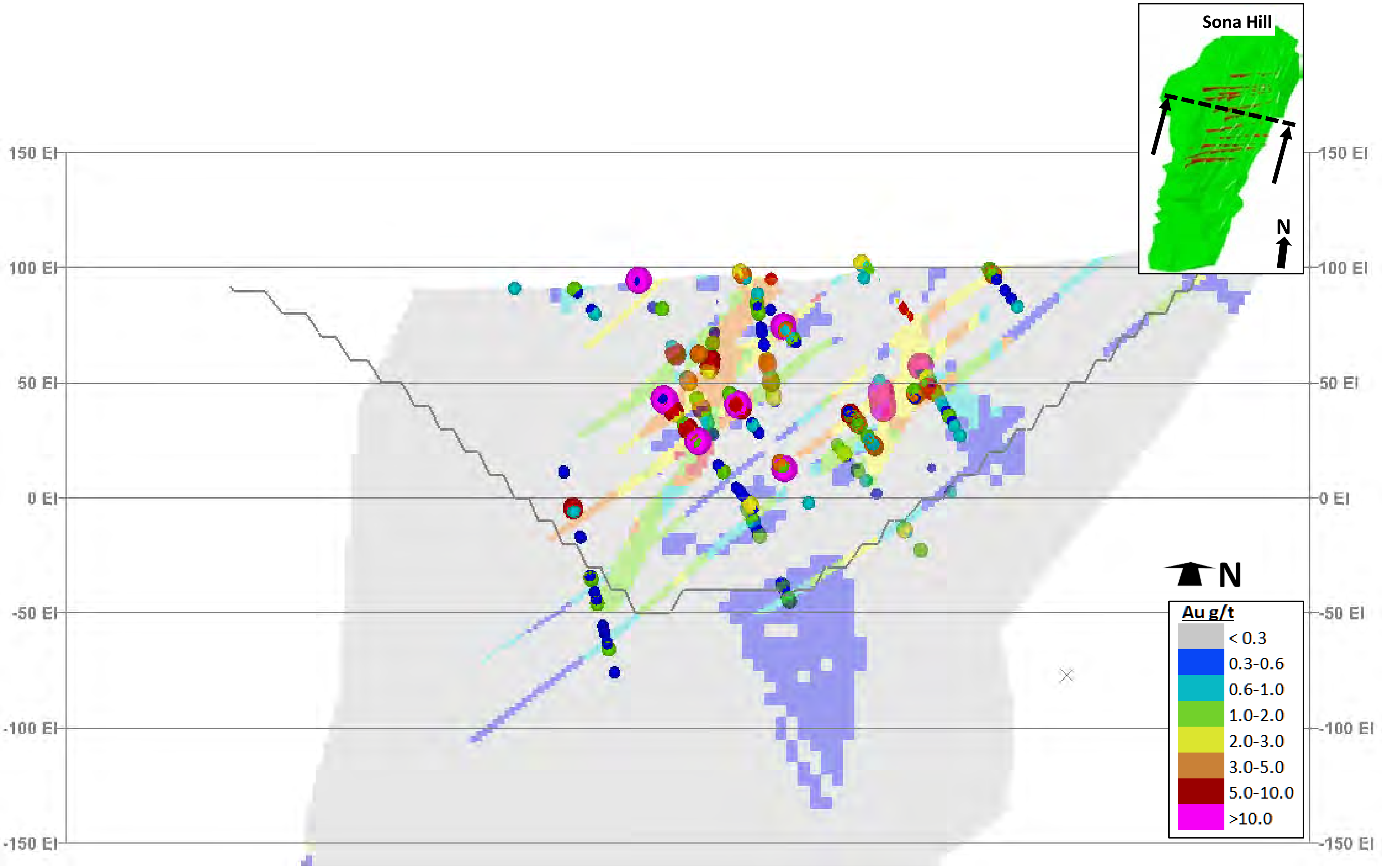
Validation: Toroparu Deposit, SE Area Cross-Section



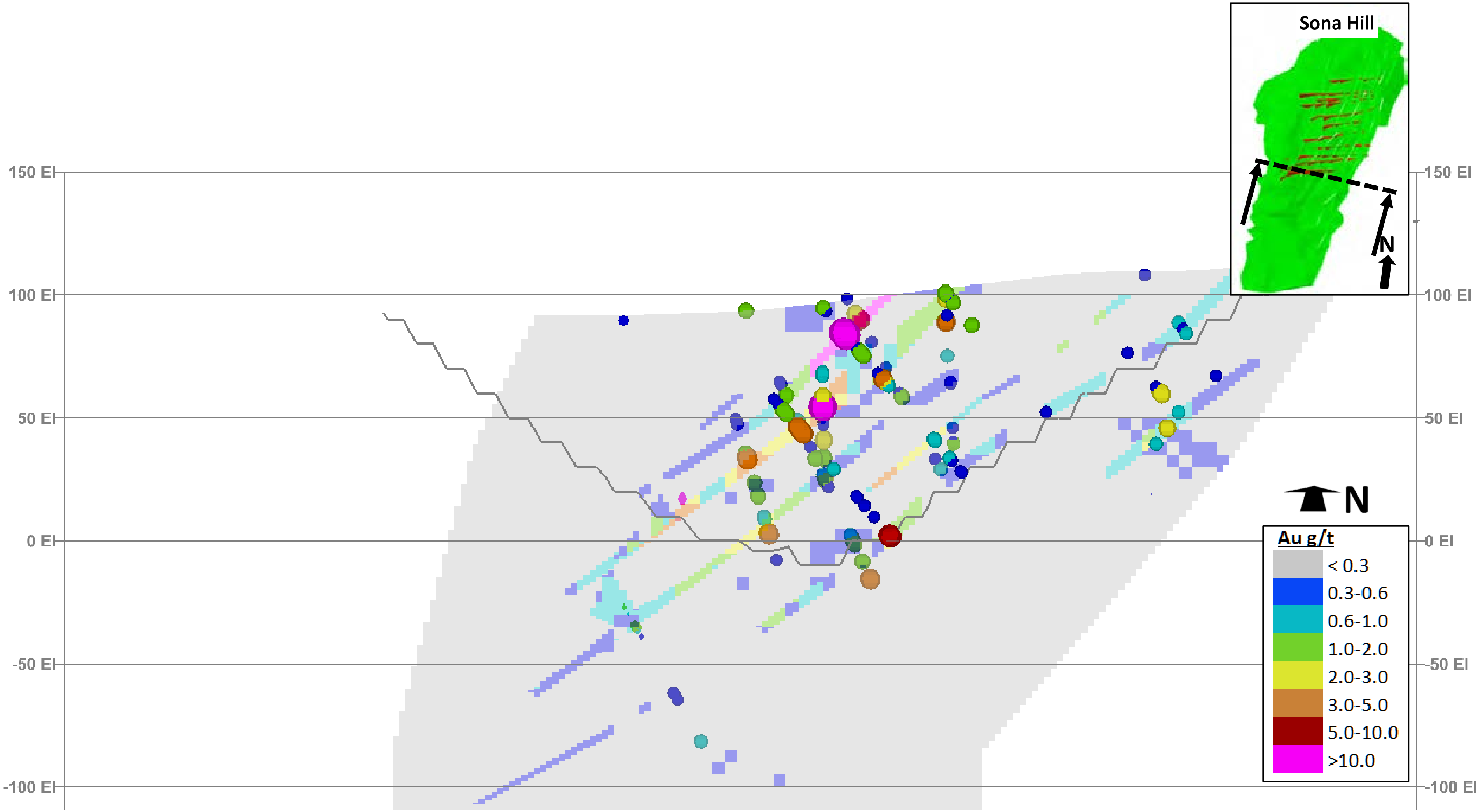
Validation: Sona Hill Deposit Cross-Section



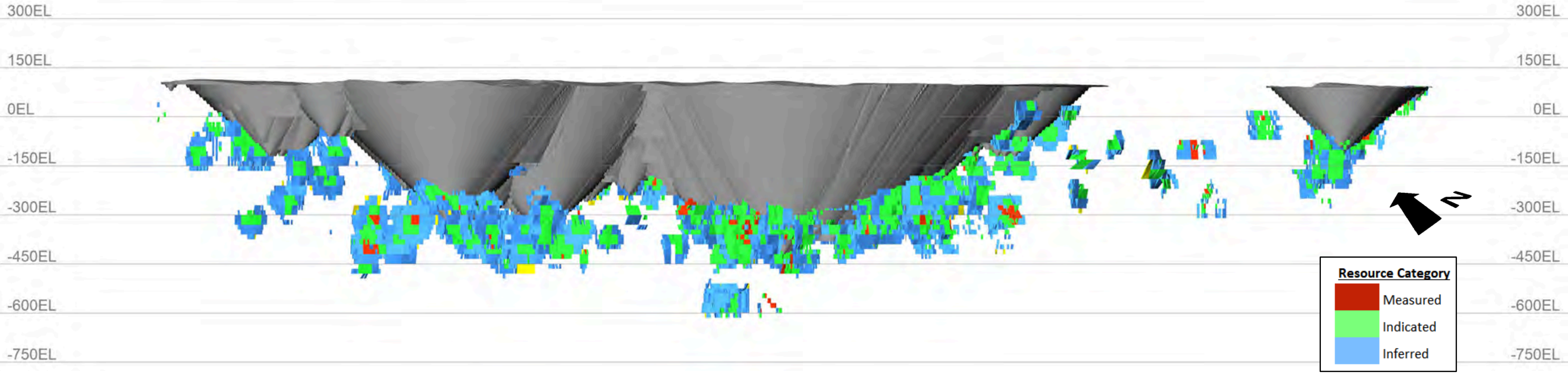
Validation: Sona Hill Deposit Cross-Section

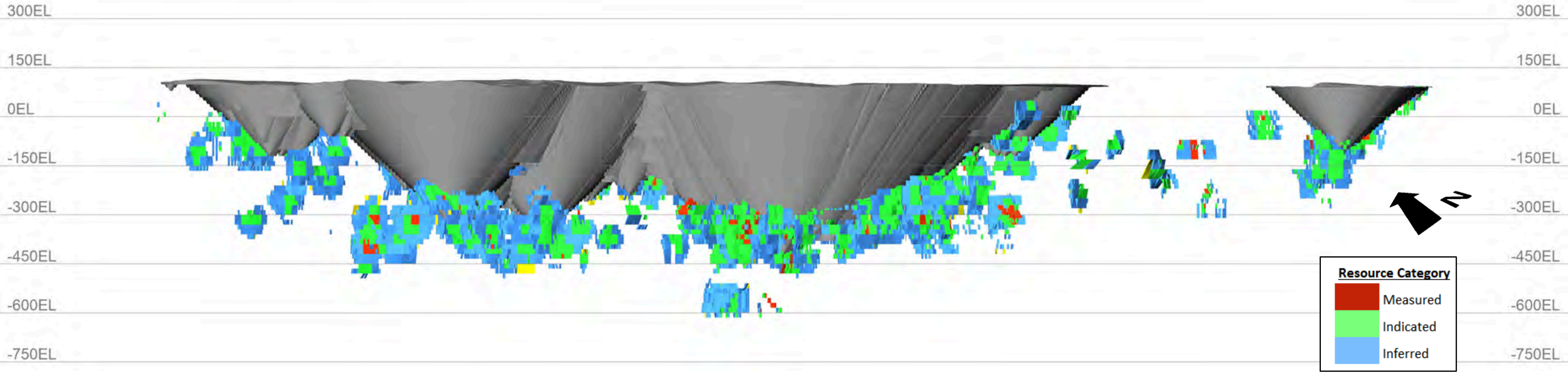


Validation: Sona Hill Deposit Cross-Section

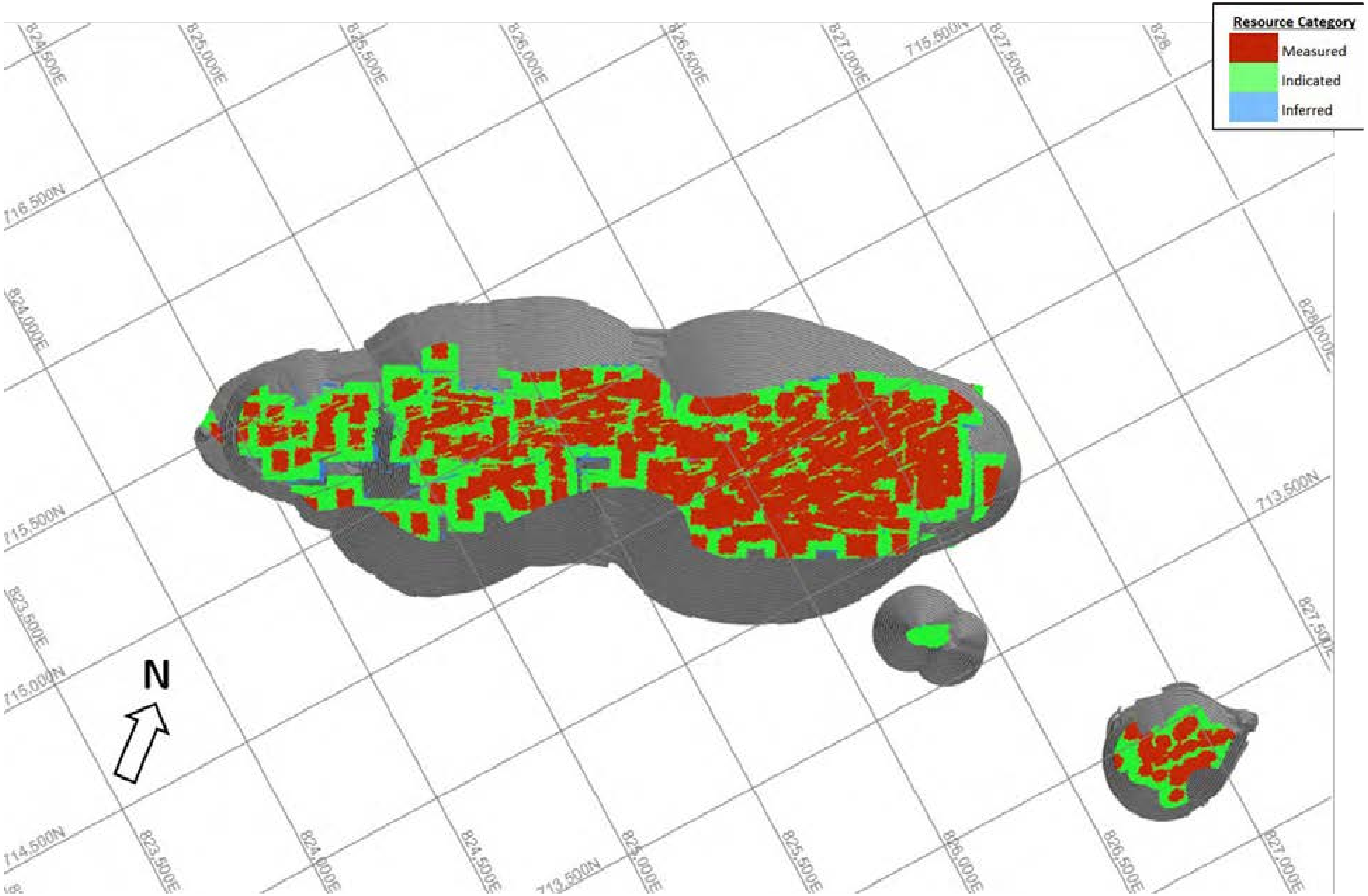


Appendix D: Resource Category Images

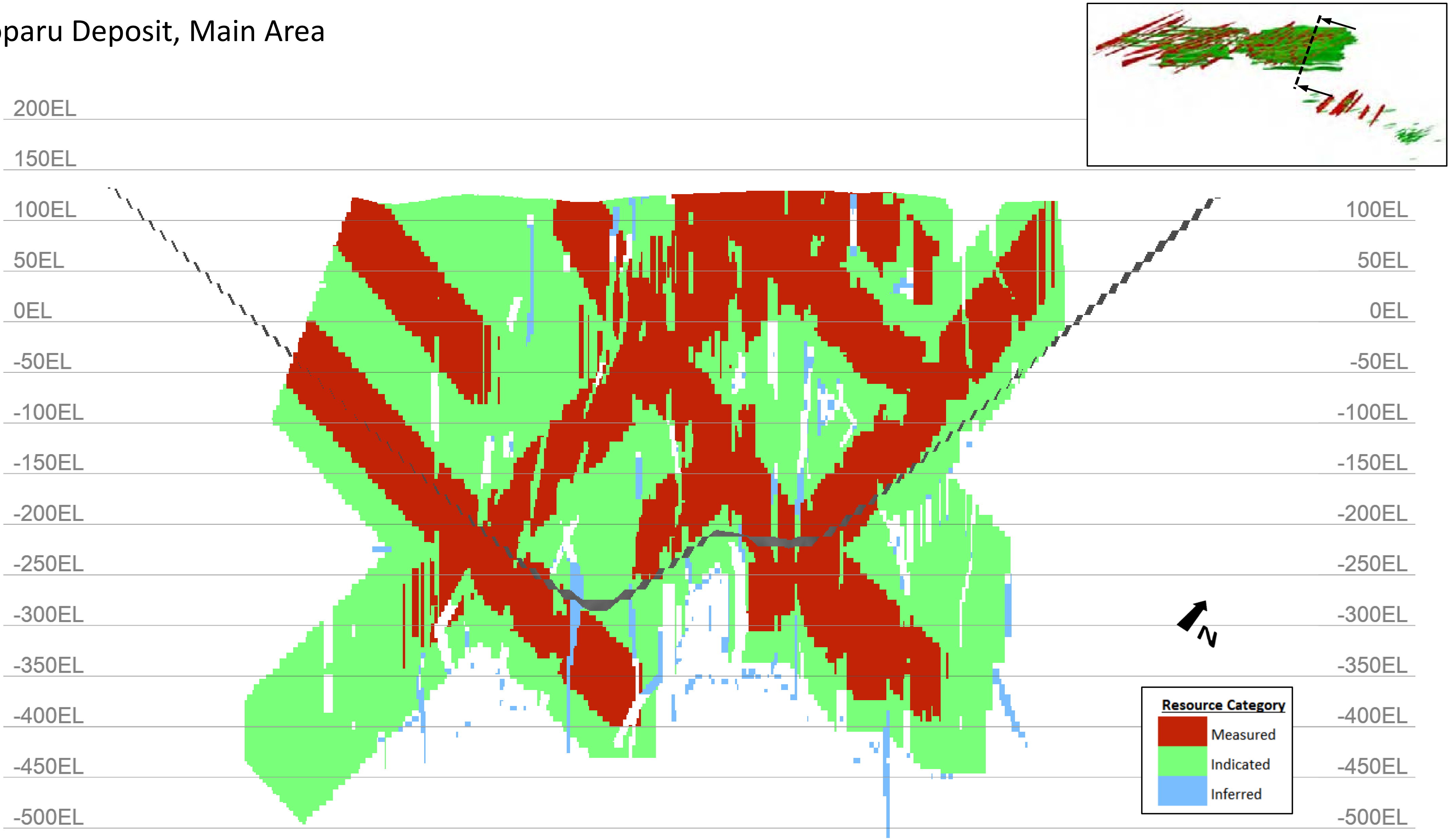

300EL
150EL
0EL
-150EL
-300EL
-450EL
-600EL
-750EL
N
Resource Category
Measured
Indicated
Inferred


300EL
150EL
0EL
-150EL
-300EL
-450EL
-600EL
-750EL
300EL
150EL
0EL
-150EL
-300EL
-450EL
-600EL
-750EL
N
Resource Category
Measured
Indicated
Inferred

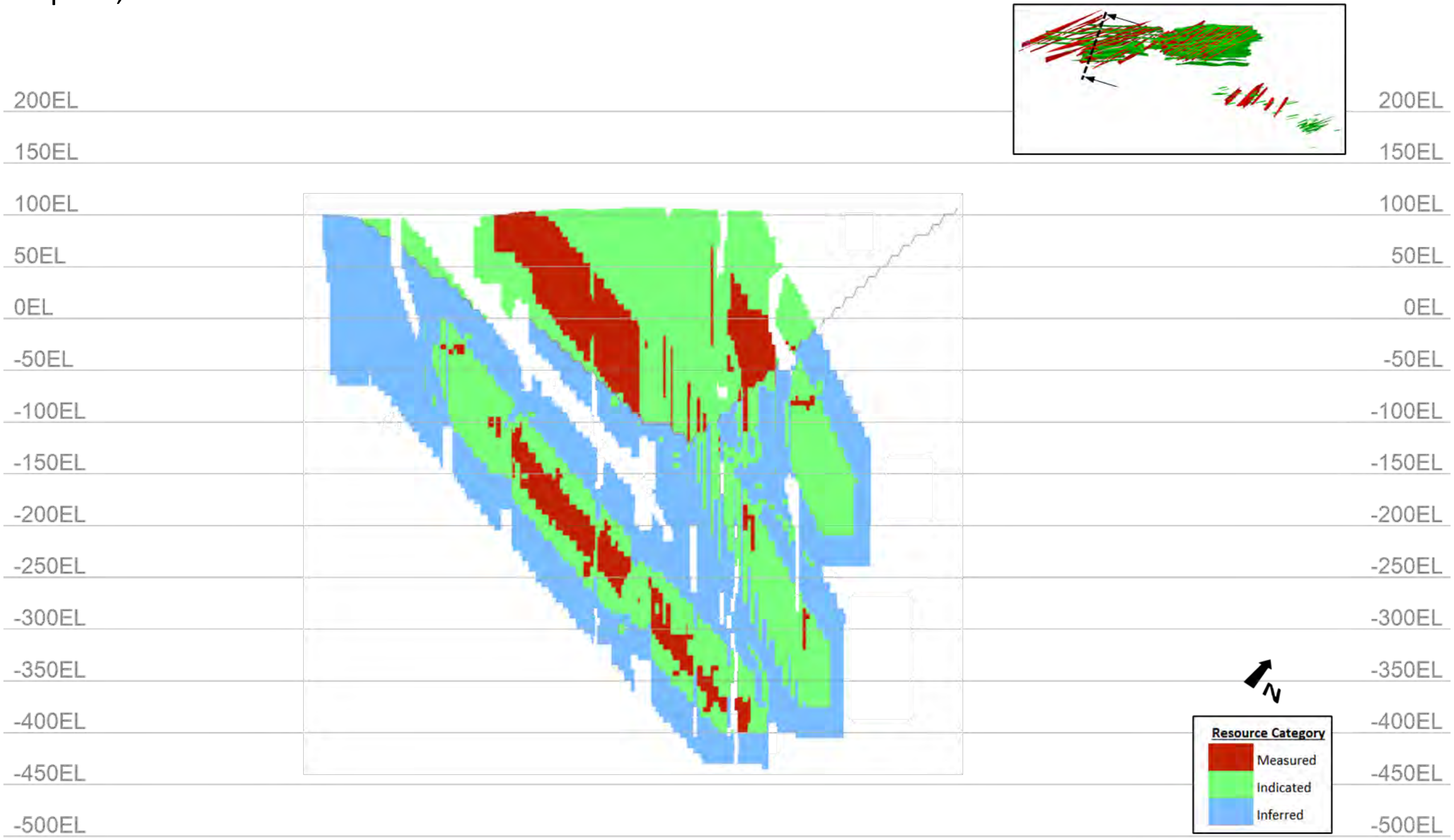
Main/NW Open Pit Resource Category



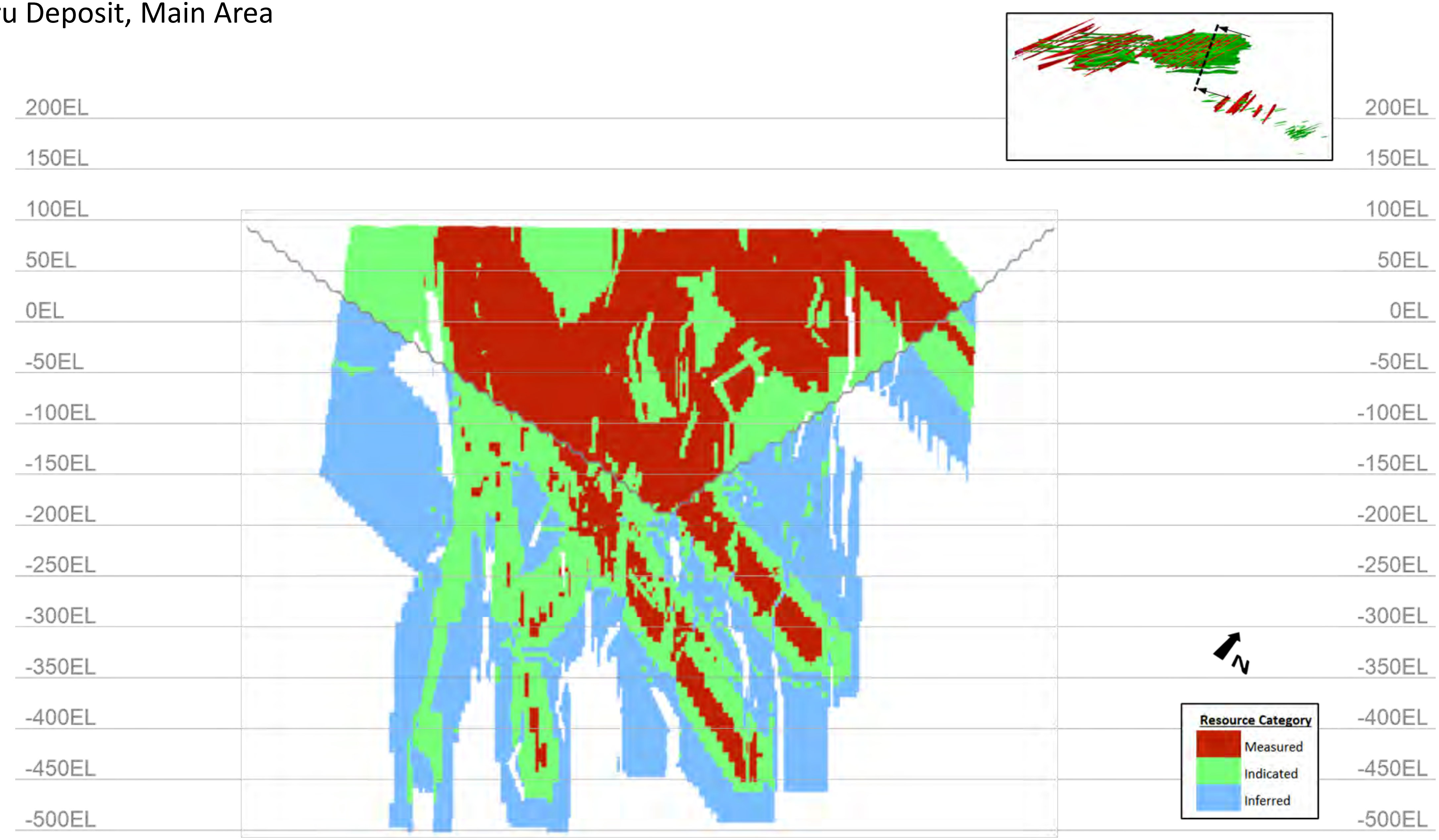
Toroparu Deposit, Main Area



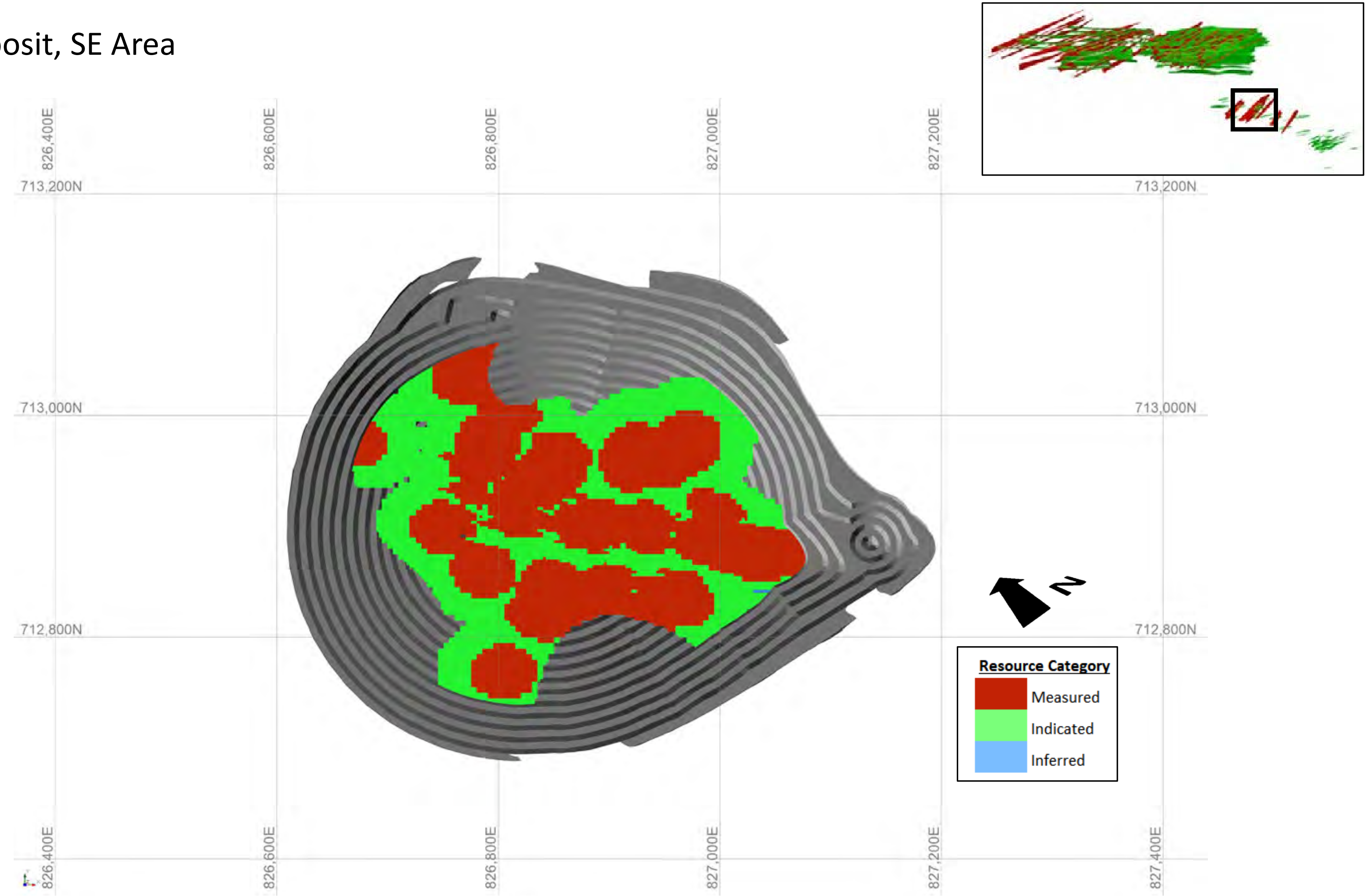
Toroparu Deposit, NW Area



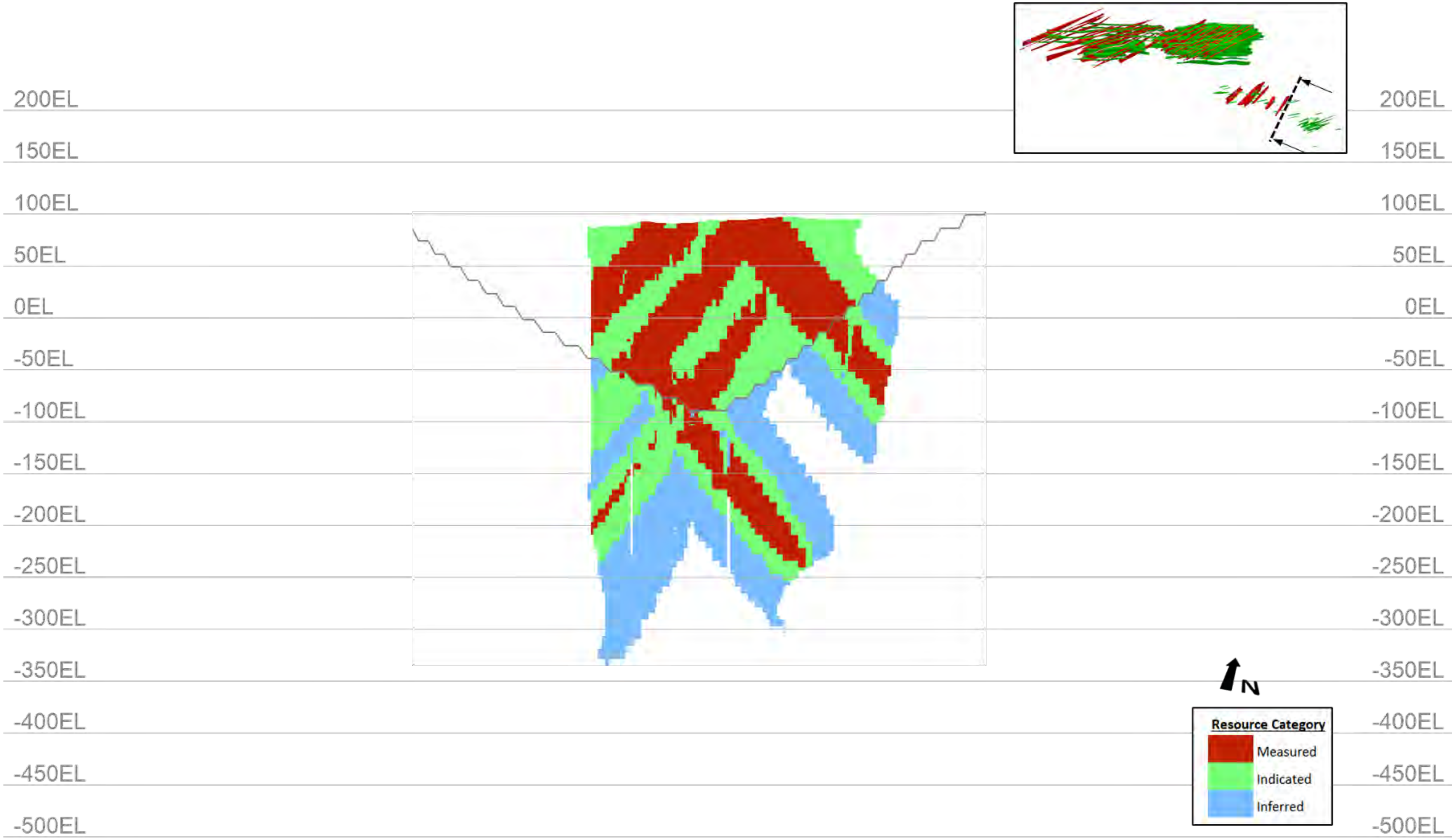
Toroparu Deposit, Main Area



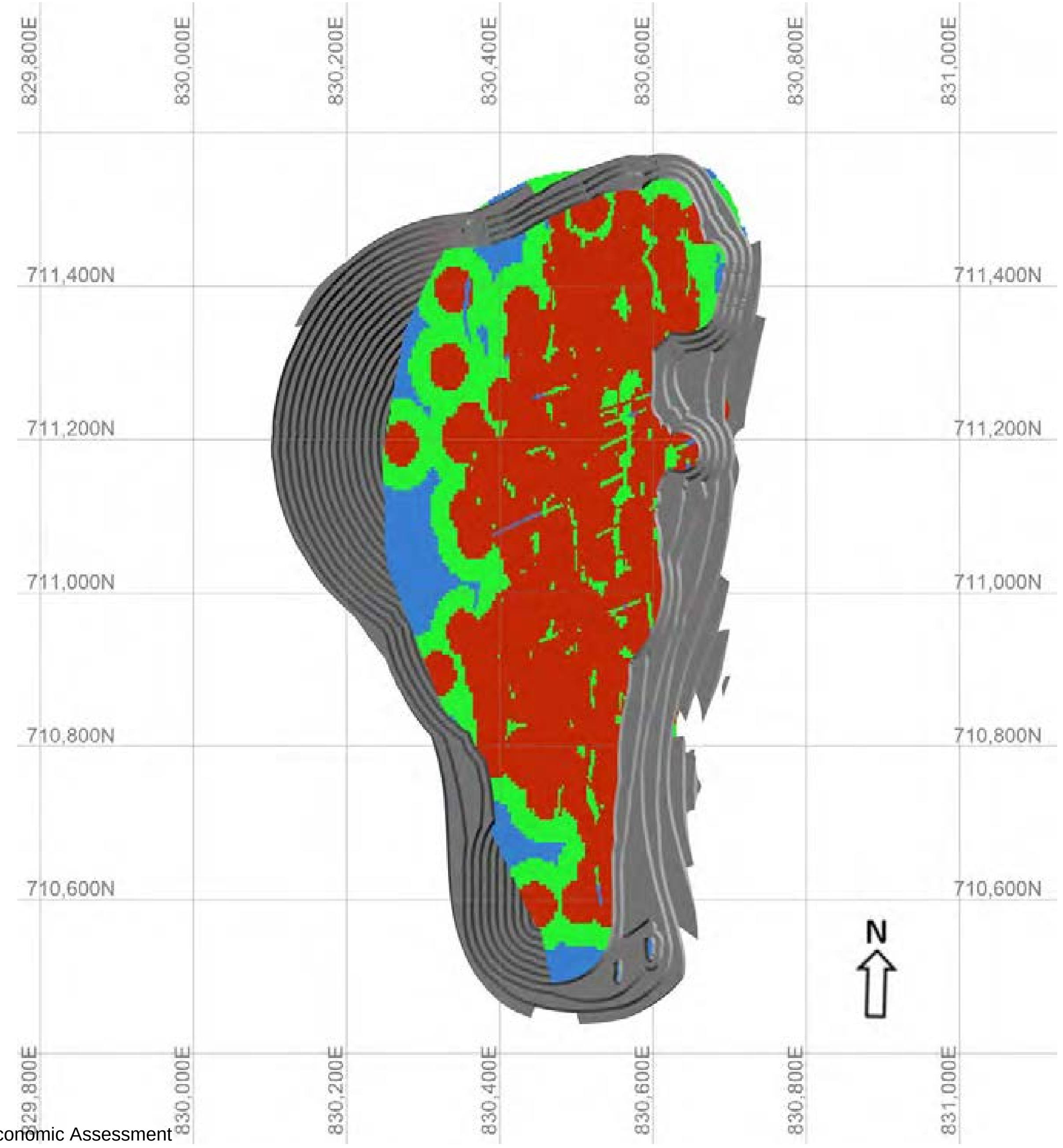
Toroparu Deposit, SE Area



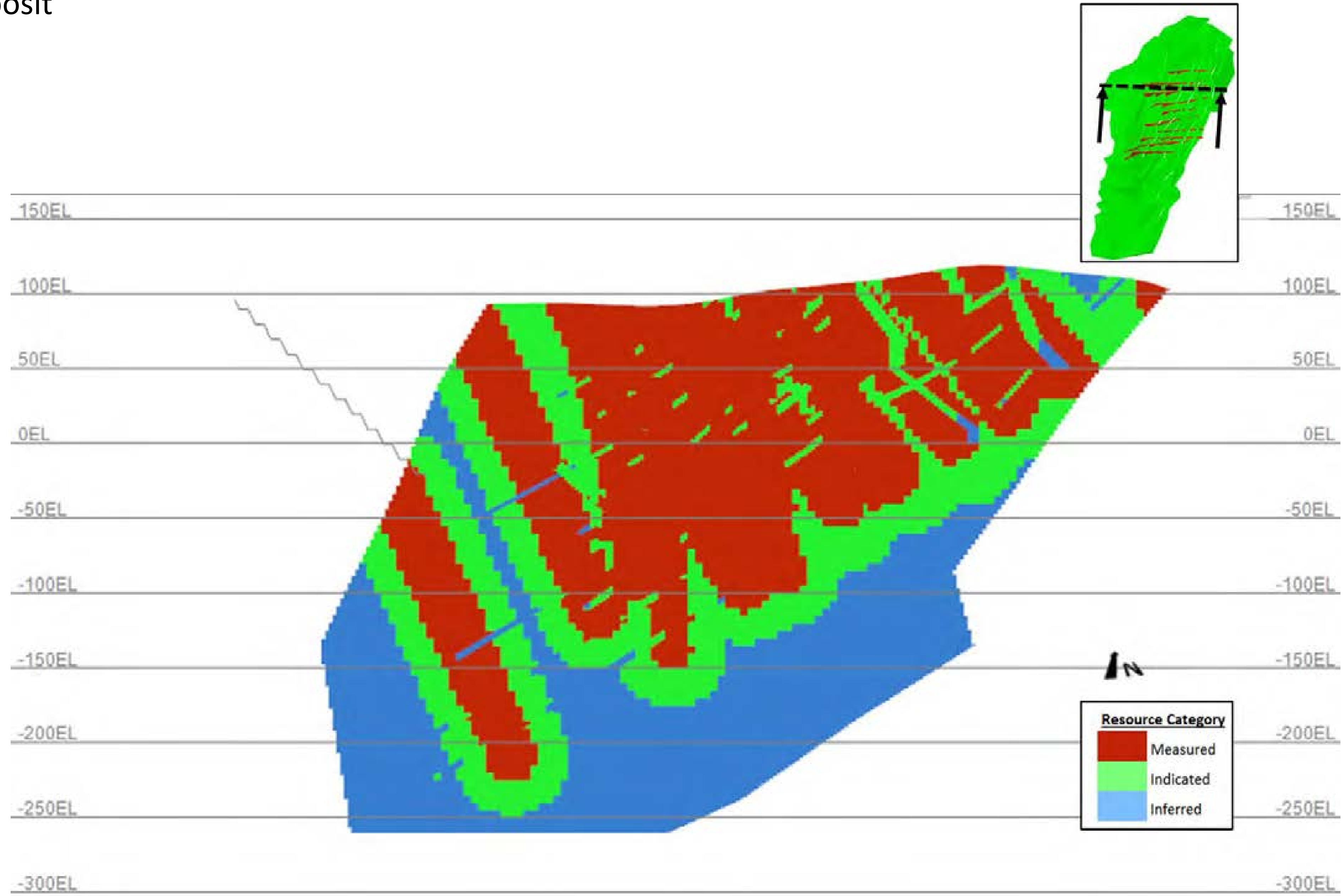
Toroparu Deposit, SE Area



Sona Hill Deposit



Sona Hill Deposit



Appendix E: Financial Model

	value /	units /	Total	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
	factor	sensit.	or Avg.	-3	-2	-1	1	2	3	4	5	6	7	8
PRODUCTION SUMMARY														
Mine Movement														
Total Waste	-	kt	557,943	-	-	14,891	17,355	17,924	15,774	17,596	17,235	17,838	18,506	22,602
SAP Ore	-	kt	9,132	-	-	1,959	1,742	2,116	434	260	699	589	290	-
CIL Ore	-	kt	45,970	-	-	244	1,268	1,256	1,632	3,094	3,329	1,980	460	1,032
Floatation Ore	-	kt	12,361	-	-	0	472	-	994	211	97	414	1,239	625
Flex Ore	-	kt	39,839	-	-	11	731	1	1,624	232	632	1,053	2,311	2,409
Total RoM	**-**	**kt**	**107,302**	**-**	**-**	**2,214**	**4,212**	**3,373**	**4,684**	**3,796**	**4,757**	**4,036**	**4,300**	**4,066**
Total Mine Movement	**-**	**kt**	**665,246**	**-**	**-**	**17,106**	**21,567**	**21,296**	**20,458**	**21,393**	**21,992**	**21,873**	**22,806**	**26,668**
Material Rehandling	-	kt	29,109	-	-	-	379	626	497	396	21	1,823	1,695	1,814
Total Movement		**kt**	**694,355**	-	-	17,106	21,945	21,923	20,955	21,789	22,013	23,696	24,501	28,482
Flotation Plant Feed														
Ore Feed to Flotation		kt	46,981	-	-	-	-	-	-	-	-	2,172	2,555	2,555
Total Flotation Feed		**kt**	**46,981**	-	-	-	-	-	-	-	-	2,172	2,555	2,555
Flotation Plant Feed Rate														
Ore Feed to Flotation		tpd	6,770	-	-	-	-	-	-	-	-	5,950	7,000	7,000
Total Flotation Feed Rate		**tpd**	**6,770**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**5,950**	**7,000**	**7,000**
CIL Feed														
Main/SE Saprolite Feed to CIL	-	kt	9,132	-	-	-	653	767	767	548	384	383	383	383
Main/SE Fresh Ore Feed to CIL	-	kt	43,882	-	-	-	1,524	953	1,186	439	1,122	952	1,163	1,156
SH Saprolite Feed to CIL	-	kt	0	-	-	-	-	-	-	-	-	-	-	-
SH Fresh Ore Feed to CIL	-	kt	7,307	-	-	-	-	835	602	1,569	1,056	1,220	1,009	1,016
Total CIL Feed		**kt**	**60,322**	-	-	-	2,178	2,555	2,555	2,555	2,562	2,555	2,555	2,555
CIL Feed Rate														
Main/SE Saprolite Feed to CIL	-	tpd	1,191	-	-	-	1,785	2,100	2,100	1,500	1,050	1,050	1,050	1,050
Main/SE Fresh Ore Feed to CIL	-	tpd	5,006	-	-	-	4,165	2,611	3,250	1,202	3,065	2,608	3,185	3,166
SH Saprolite Feed to CIL	-	tpd	0	-	-	-	-	-	-	-	-	-	-	-
SH Fresh Ore Feed to CIL	-	tpd	2,859	-	-	-	-	2,289	1,650	4,298	2,885	3,342	2,765	2,784
Total CIL Feed Rate		**tpd**	**6,881**	**-**	**-**	**-**	**5,950**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**
Cumulative Process Head Grade														
Cumulative CIL Feed Au	**-**	g/t	**2.16**	-	-	-	4.18	3.54	3.25	3.14	3.01	2.75	2.54	2.35
Cumulative Flotation Feed Au	**-**	g/t	**1.11**	-	-	-	-	-	-	-	-	0.98	0.96	1.02
Cumulative Au Grade	**-**	g/t	**1.70**	-	-	-	4.18	3.54	3.25	3.14	3.01	2.53	2.20	1.99
Cumulative Flotation Feed Cu	**-**	**%**	**0.17%**	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.21%	0.21%	0.20%
Production														
Gold in Dore	-	koz	4,337	-	-	-	280	228	205	207	189	126	114	101
Gold in Concentrate	-	koz	1,106	-	-	-	-	-	-	-	-	47	53	61
Total Recovered Gold	**0.883573268**	**koz**	**5,443**	**-**	**-**	**-**	**280**	**228**	**205**	**207**	**189**	**173**	**167**	**162**
Cumulative Recovered Gold				**-**	**-**	**-**	**280**	**508**	**713**	**921**	**1,110**	**1,283**	**1,450**	**1,612**
Cumulative avg. Annual Gold Produced		**kozs/y**	**233**		**-**		**280**	**254**	**238**	**230**	**222**	**214**	**207**	**201**
Copper Concentrate	-	kt	311	-	-	-	-	-	-	-	-	18	21	17
Copper in Concentrate		kt	67.2	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**3.8**	**4.6**	**3.7**
Copper in Concentrate	-	klb	148,141	-	-	-	-	-	-	-	-	8,399	10,092	8,069
Payable Gold														
Payable Gold in Dore	-	koz	4,335	-	-	-	280	228	204	207	189	126	114	101
Payable Gold in Concentrate	-	koz	1,073	-	-	-	-	-	-	-	-	46	52	60
Total Payable Gold	**-**	**koz**	**5,407**	**-**	**-**	**-**	**280**	**228**	**204**	**207**	**189**	**171**	**165**	**160**
Sold at Spot		**koz**	**4,867**	**-**	-	-	252	205	184	187	170	154	149	144
Sold to SLW		**koz**	**541**	**-**	-	-	28	23	20	21	19	17	17	16
Payable Silver														
Payable Silver in Dore	-	koz	1,292	-	-	-	70	57	104	63	24	72	111	60
Paynle Silver in Concentrate	-	koz	1,230	-	-	-	-	-	-	-	-	80	101	76
Total Payable Silver	**-**	**koz**	**2,522**	**-**	-	-	70	57	104	63	24	152	212	136
Sold at Spot		**koz**	**1,261**	**-**	-	-	35	28	52	32	12	76	106	68
Sold to SLW		**koz**	**1,261**	**-**	-	-	35	28	52	32	12	76	106	68
Payable Copper	**-**	**klb**	**141,295**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**8,012**	**9,628**	**7,695**
Payable Copper	-	t	64,090											
Tailings														
Flotation Tailings	-	kt	46,670	-	-	-	-	-	-	-	-	2,154	2,534	2,538
CIL Tailings	-	kt	60,322	-	-	-	2,178	2,555	2,555	2,555	2,562	2,555	2,555	2,555
Total Tailings	**-**	**kt**	**106,992**	**-**	**-**	**-**	**2,178**	**2,555**	**2,555**	**2,555**	**2,562**	**4,709**	**5,089**	**5,093**

	value / factor	units / sensit.	Total or Avg.	2021 -3	2022 -2	2023 -1	2024 1	2025 2	2026 3	2027 4	2028 5	2029 6	2030 7	2031 8
CASH FLOW SCHEDULE														
Price Schedule														
Gold - Sold at Spot		US$/oz	-	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500
Gold - Sold to SLW		US$/oz	-	400	400	400	400	400	400	400	400	400	400	400
Silver - Sold at Spot		US$/oz	-	20	20	20	20	20	20	20	20	20	20	20
Silver - Sold to SLW		US$/oz	-	4	4	4	4	4	4	4	4	4	4	4
Copper		US$/lb	-	3.13	3.13	3.13	3.13	3.13	3.13	3.13	3.13	3.13	3.13	3.13
Net Revenue Before By-Product Credits														
Gold	94%	$000s	7,516,386	-	-	-	389,162	317,268	284,236	288,226	262,974	238,172	229,673	223,035
Silver	0%	$000s	30,413	-	-	-	840	683	1,250	764	285	1,833	2,556	1,644
Copper	6%	$000s	441,660	-	-	-	0	0	0	0	0	25,045	30,094	24,053
Total Net Revenue Before By-Product Credits	-	**$000s**	**7,988,460**	**-**	**-**	**-**	**390,002**	**317,951**	**285,487**	**288,990**	**263,259**	**265,051**	**262,322**	**248,732**
Net Revenue														
Gold	-	$000s	7,516,386	-	-	-	389,162	317,268	284,236	288,226	262,974	238,172	229,673	223,035
Silver	-	$000s	0	-	-	-	0	0	0	0	0	0	0	0
Copper	-	$000s	0	-	-	-	0	0	0	0	0	0	0	0
Total Net Revenue	-	**$000s**	**7,516,386**	**-**	**-**	**-**	**389,162**	**317,268**	**284,236**	**288,226**	**262,974**	**238,172**	**229,673**	**223,035**
Direct Cash Costs	($4.14)													
Mining Cost (OP & UG)	($17.16)	$000s	(1,840,872)	-	-	-	(42,460)	(42,754)	(42,177)	(44,693)	(44,188)	(40,967)	(45,021)	(53,647)
Processing Cost	($14.52)	$000s	(1,557,511)	-	-	-	(34,541)	(38,555)	(38,155)	(38,976)	(39,728)	(69,404)	(73,720)	(73,662)
Site G&A Cost	($3.36)	$000s	(360,096)	-	-	-	(14,434)	(14,385)	(14,684)	(15,120)	(15,455)	(15,886)	(16,651)	(16,656)
Freight Cost	($0.43)	$000s	(46,297)	-	-	-	0	0	0	0	0	(2,614)	(3,138)	(2,528)
Treatment Charges	($0.17)	$000s	(18,634)	-	-	-	0	0	0	0	0	(1,052)	(1,263)	(1,017)
Refining Charges	($0.17)	$000s	(18,019)	-	-	-	(269)	(219)	(196)	(199)	(182)	(843)	(965)	(861)
Predicted Penalties	($0.01)	$000s	(1,553)	-	-	-	0	0	0	0	0	(88)	(105)	(85)
By-Product Credit	$4.40	$000s	472,073	-	-	-	840	683	1,250	764	285	26,879	32,650	25,697
Total Direct Cash Costs	1	**$000s**	**(3,370,909)**	**-**	**-**	**-**	**(90,863)**	**(95,230)**	**(93,961)**	**(98,224)**	**(99,267)**	**(103,974)**	**(108,213)**	**(122,760)**
Direct Cash Cost	**-**	**US$/oz-Au**	**623**	**-**	**-**	**-**	**325**	**417**	**459**	**474**	**525**	**607**	**655**	**765**
Indirect Cash Costs														
Royalties	($5.81)	$000s	(623,952)	**-**	**-**	**-**	**(31,109)**	**(27,361)**	**(24,761)**	**(25,049)**	**(23,004)**	**(21,304)**	**(20,720)**	**(20,019)**
Freight Insurance Cost	-	$000s	(17,211)	**-**	**-**	**-**	**(876)**	**(714)**	**(773)**	**(678)**	**(533)**	**(639)**	**(728)**	**(575)**
Total Indirect Cash Costs		**$000s**	**(641,163)**	**-**	**-**	**-**	**(31,984)**	**(28,075)**	**(25,534)**	**(25,727)**	**(23,536)**	**(21,942)**	**(21,448)**	**(20,594)**
Indirect Cash Cost		**US$/oz-Au**	**119**	**-**	**-**	**-**	**114**	**123**	**125**	**124**	**124**	**128**	**130**	**128**
Sustaining Cash Cost		**US$/oz-Au**	**175**	**-**	**-**	**-**	**41**	**146**	**120**	**451**	**993**	**81**	**64**	**165**
AISC Cost		**US$/oz-Au**	**916**	**-**	**-**	**-**	**480**	**686**	**704**	**1,049**	**1,642**	**816**	**849**	**1,058**
Cumulative AISC							***480***	***572***	***610***	***709***	***868***	***861***	***860***	***880***
Total Operating Expense	-	**$000s**	**(4,012,072)**	**-**	**-**	**-**	**(122,847)**	**(123,305)**	**(119,495)**	**(123,952)**	**(122,804)**	**(125,917)**	**(129,661)**	**(143,354)**
Operating Margin	-	**$000s**	**3,504,315**	**-**	**-**	**-**	**266,314**	**193,963**	**164,741**	**164,274**	**140,170**	**112,256**	**100,011**	**79,681**
Cash Available for Debt Service														
Operating Margin		$000s	3,504,315	-	-	-	266,314	193,963	164,741	164,274	140,170	112,256	100,011	79,681
Project Capital	**(354,760)**	$000s	(1,298,570)	(3,474)	(127,927)	(223,359)	(11,517)	(33,243)	(24,489)	(93,506)	(187,891)	(13,833)	(10,586)	(26,428)
Income Tax	-	$000s	(649,572)	-	-	-	(38,418)	(34,218)	(23,982)	(18,231)	0	(12,226)	(10,185)	(3,970)
Working Capital	-	$000s	0	-	-	-	(18,287)	(852)	136	(755)	(116)	(5,377)	(1,827)	(1,715)
Cash Flow	**(354,760)**	**$000s**	**1,556,172**	**(3,474)**	**(127,927)**	**(223,359)**	**198,092**	**125,650**	**116,407**	**51,782**	**(47,837)**	**80,820**	**77,413**	**47,569**
GPA Installment Payments	138,000	$000s	138,000	-	49,763	88,237	0	0	0	0	0	0	0	0
Loan Repayment	-	$000s	0	-	-	-	0	0	0	0	0	0	0	0
Interest Expense	-	$000s	0	-	-	-	0	0	0	0	0	0	0	0
Free Cash Flow	**(216,760)**	**$000s**	**1,694,172**	**(3,474)**	**(78,164)**	**(135,122)**	**198,092**	**125,650**	**116,407**	**51,782**	**(47,837)**	**80,820**	**77,413**	**47,569**
ECONOMICS METRICS														
Cumulative FCF			**1.15**	**(3,474)**	**(81,638)**	**(216,760)**	**(18,667)**	**106,983**	**223,389**	**275,171**	**227,334**	**308,154**	**385,567**	**433,136**
After-Tax IRR			**46.08%**	**12**	**12**	**12**	**12**	**2**	**0**	**0**	**0**	**0**	**0**	**0**
NPV Period				**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**
After-Tax Present Value	0%	$000s	**1,694,172**	**(3,474)**	**(78,164)**	**(135,122)**	**198,092**	**125,650**	**116,407**	**51,782**	**(47,837)**	**80,820**	**77,413**	**47,569**
After-Tax Present Value	3%	$000s	**1,058,018**	**(3,474)**	**(75,887)**	**(127,365)**	**181,283**	**111,638**	**100,413**	**43,367**	**(38,896)**	**63,800**	**59,330**	**35,396**
After-Tax Present Value	5%	$000s	**794,034**	**(3,474)**	**(74,442)**	**(122,559)**	**171,120**	**103,373**	**91,208**	**38,641**	**(33,997)**	**54,702**	**49,901**	**29,203**
After-Tax Present Value	8%	$000s	**535,423**	**(3,474)**	**(72,374)**	**(115,845)**	**157,252**	**92,357**	**79,224**	**32,631**	**(27,912)**	**43,664**	**38,726**	**22,034**
After-Tax Present Value	10%	$000s	**420,676**	**(3,474)**	**(71,058)**	**(111,671)**	**148,830**	**85,821**	**72,279**	**29,230**	**(24,548)**	**37,703**	**32,831**	**18,340**
Cumulative After-Tax Present Value	5%	$000s	-	**(3,474)**	**(77,916)**	**(200,475)**	**(29,356)**	**74,017**	**165,225**	**203,865**	**169,868**	**224,570**	**274,471**	**303,674**
Cumulative After-Tax Present Value	8%	$000s	-	**(3,474)**	**(75,848)**	**(191,693)**	**(34,441)**	**57,915**	**137,140**	**169,771**	**141,859**	**185,523**	**224,249**	**246,282**
Cumulative After-Tax Present Value	10%	$000s	-	**(3,474)**	**(74,532)**	**(186,203)**	**(37,373)**	**48,447**	**120,727**	**149,956**	**125,408**	**163,111**	**195,942**	**214,282**
			0.92				**0.916492357**							
Pre-Tax IRR			**58.53%**	**12**	**12**	**12**	**10.99790828**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
Pre-Tax Present Value	0%		**2,343,744**	**(3,474)**	**(78,164)**	**(135,122)**	**236,510**	**159,868**	**140,388**	**70,013**	**(47,837)**	**93,045**	**87,598**	**51,539**
Pre-Tax Present Value	5%		**1,108,302**	**(3,474)**	**(74,442)**	**(122,559)**	**204,306**	**131,524**	**109,998**	**52,245**	**(33,997)**	**62,977**	**56,466**	**31,640**
Pre-Tax Present Value	8%		**754,482**	**(3,474)**	**(72,374)**	**(115,845)**	**187,749**	**117,508**	**95,546**	**44,120**	**(27,912)**	**50,270**	**43,821**	**23,872**
EBTIDA	-		**3,504,315**	**-**	**-**	**-**	**266,314**	**193,963**	**164,741**	**164,274**	**140,170**	**112,256**	**100,011**	**79,681**
Cumulative Pre-Tax FCF				**(3,474)**	**(81,638)**	**(216,760)**	**19,750**	**179,618**	**320,006**	**390,020**	**342,183**	**435,228**	**522,826**	**574,365**
Begin Equity		$000s		-	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
GPA Installment Payments		$000s	**138,000**	**-**	**49,763**	**88,237**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Debt		$000s	**0**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Construction Cost or Cover of Losses		$000s	**(402,597)**	**(3,474)**	**(127,927)**	**(223,359)**	**-**	**-**	**-**	**-**	**(47,837)**	**-**	**-**	**-**
Additional Equity		$000s	**216,760**	**3,474**	**78,164**	**135,122**	**-**	**-**	**-**	**-**	**47,837**	**-**	**-**	**-**
End		$000s		**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Initial Equity		$000s	**0**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Additional Equity		$000s	**(264,597)**	**(3,474)**	**(78,164)**	**(135,122)**	**-**	**-**	**-**	**-**	**(47,837)**	**-**	**-**	**-**
Dividends		$000s	**1,958,769**	-	-	-	**198,092**	**125,650**	**116,407**	**51,782**	**-**	**80,820**	**77,413**	**47,569**
Net Cashflow		$000s	**1,694,172**	**(3,474)**	**(78,164)**	**(135,122)**	**198,092**	**125,650**	**116,407**	**51,782**	**(47,837)**	**80,820**	**77,413**	**47,569**
Return on Equity			**46.08%**											

	value / factor	units / sensit.	Total or Avg.	2021 -3	2022 -2	2023 -1	2024 1	2025 2	2026 3	2027 4	2028 5	2029 6	2030 7	2031 8
COPPER CONCENTRATE FREIGHT														
Freight & Marketing	$/t-conc		0.0%											
Road Transport	$12.15	$000s	(4,101)	-	-	-	0	0	0	0	0	(232)	(278)	(224)
Rail Freight	$0.00	$000s	0	-	-	-	0	0	0	0	0	0	0	0
Handling Charge	$0.00	$000s	0	-	-	-	0	0	0	0	0	0	0	0
Ocean Freight	$100.00	$000s	(33,756)	-	-	-	0	0	0	0	0	(1,906)	(2,288)	(1,843)
Container Discharge Fee	$25.00	$000s	(8,439)	-	-	-	0	0	0	0	0	(476)	(572)	(461)
Subtotal		**$000s**	**(46,297)**	**-**	**-**	**-**	**0**	**0**	**0**	**0**	**0**	**(2,614)**	**(3,138)**	**(2,528)**
ROYALTIES														
Royalties														
Guyana Au Royalty	8.0%	$000s	(595,750)	-	-	-	(31,052)	(25,316)	(22,669)	(22,996)	(20,988)	(18,827)	(18,119)	(17,570)
Guyana Ag Royalty	8.0%	$000s	(2,018)	-	-	-	(56)	(46)	(92)	(53)	(16)	(126)	(180)	(112)
Guyana Cu Royalty	1.5%	$000s	(6,185)	-	-	-	0	0	0	0	0	(351)	(422)	(337)
One Time Royalty to Surface Owner	-	$000s	(20,000)	-	-	-	0	(2,000)	(2,000)	(2,000)	(2,000)	(2,000)	(2,000)	(2,000)
Total Royalties		**$000s**	**(623,952)**	**-**	**-**	**-**	**(31,109)**	**(27,361)**	**(24,761)**	**(25,049)**	**(23,004)**	**(21,304)**	**(20,720)**	**(20,019)**
PROJECT CAPITAL			LoM											
Project Capital														
Pre-Stripping Capex - Mining Costs		$000s	31,774	-	-	31,774	0	0	0	0	0	0	0	0
Pre-Stripping Capex - G&A Costs		$000s	9,242	-	-	9,242	0	0	0	0	0	0	0	0
Mine (OP & UG)		$000s	694,913	-	11,047	13,160	10,290	12,995	24,489	10,975	58,233	13,833	10,586	26,428
Process Plant		$000s	198,220	-	30,457	64,663	0	0	0	32,883	70,218	0	0	0
Water and Tailings Management		$000s	45,299	-	8,899	7,697	0	13,727	0	0	0	0	0	0
Infrastructure		$000s	69,411	3,474	31,690	28,556	1,227	4,463	0	0	0	0	0	0
Power Supply		$000s	3,228	-	957	2,271	0	0	0	0	0	0	0	0
Owner's		$000s	66,402	-	6,238	16,592	0	0	0	10,678	10,678	0	0	0
Indirect Costs		$000s	112,832	-	26,070	26,070	0	0	0	30,346	30,346	0	0	0
Risk and Contingency		$000s	67,249	-	12,570	23,333	0	2,059	0	8,624	18,416	0	0	0
Total Capital	**1**	**354,760**	**1,298,570**	**3,474**	**127,927**	**223,359**	**11,517**	**33,243**	**24,489**	**93,506**	**187,891**	**13,833**	**10,586**	**26,428**
Cum. Capital				**3,474**	**131,401**	**354,760**	**366,277**	**399,521**	**424,009**	**517,515**	**705,406**	**719,239**	**729,825**	**756,253**
Initial Capital			**354,760**	**3,474**	**127,927**	**223,359**								
Sustaining Capital			**943,811**				**11,517**	**33,243**	**24,489**	**93,506**	**187,891**	**13,833**	**10,586**	**26,428**
Working Capital														
Beginning Balance	-	$000s	751,696	-	-	-	0	18,287	19,139	19,003	19,758	19,874	25,251	27,079
Ending Balance	**20%**	$000s	751,696	-	-	-	18,287	19,139	19,003	19,758	19,874	25,251	27,079	28,793
Change	-	**$000s**	**0**	**-**	**-**	**-**	**(18,287)**	**(852)**	**136**	**(755)**	**(116)**	**(5,377)**	**(1,827)**	**(1,715)**
INCOME TAX			**LoM**	**-3**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**
Income Tax														
Net Revenue	-	$000s	7,516,386	-	-	-	389,162	317,268	284,236	288,226	262,974	238,172	229,673	223,035
Operating Expenses	-	$000s	(4,012,072)	-	-	-	(122,847)	(123,305)	(119,495)	(123,952)	(122,804)	(125,917)	(129,661)	(143,354)
Operating Profit	-	**$000s**	**3,504,315**	**-**	**-**	**-**	**266,314**	**193,963**	**164,741**	**164,274**	**140,170**	**112,256**	**100,011**	**79,681**
Interest Expense	-	$000s	0	-	-	-	0	0	0	0	0	0	0	0
Depreciation	-	$000s	(1,298,570)	-	-	-	(73,255)	(79,904)	(84,802)	(103,503)	(141,081)	(70,592)	(66,061)	(66,449)
Net Income	-	**$000s**	**2,205,744**	**-**	**-**	**-**	**193,059**	**114,059**	**79,939**	**60,771**	**(911)**	**41,663**	**33,950**	**13,232**
Loss Carry Forward			0											
Additions	-	**$000s**	**25,407**	**-**	**-**	**-**	**0**	**0**	**0**	**0**	**911**	**0**	**0**	**0**
Opening Balance	-	$000s		**65,000**	65,000	65,000	65,000	0	0	0	911	911	0	0
Losses Used	-	$000s	65,911	-	-	-	65,000	0	0	0	0	911	0	0
Closing Balance	-	$000s	509,794	65,000	65,000	65,000	0	0	0	0	911	0	0	0
Loss Carry Forward	-	$000s	(40,504)	-	-	-	(65,000)	0	0	0	911	(911)	0	0
Taxable Income	-	**$000s**	**2,165,241**	**-**	**-**	**-**	**128,059**	**114,059**	**79,939**	**60,771**	**0**	**40,752**	**33,950**	**13,232**
Effective Income Tax	**30%**	**$000s**	**649,572**	**-**	**-**	**-**	**38,418**	**34,218**	**23,982**	**18,231**	**0**	**12,226**	**10,185**	**3,970**

	value / factor	units / sensit.	Total or Avg.	2032 9	2033 10	2034 11	2035 12	2036 13	2037 14	2038 15	2039 16	2040 17	2041 18	2042 19	2043 20	2044 21	2045 22	2046 23	2047 24	2048 25
PRODUCTION SUMMARY																				
Mine Movement																				
Total Waste	-	kt	557,943	22,012	21,957	23,318	29,589	27,673	30,157	29,918	29,573	29,943	29,259	27,767	26,309	28,245	25,110	16,352	1,038	-
SAP Ore	-	kt	9,132	80	318	33	54	109	49	300	23	-	-	77	2	-	-	-	-	-
CIL Ore	-	kt	45,970	1,235	1,290	2,669	1,617	1,696	1,713	1,855	1,707	2,069	2,502	2,464	2,635	3,515	2,520	1,842	346	-
Floatation Ore	-	kt	12,361	1,245	1,171	187	326	1,523	114	200	396	216	538	1,089	754	272	176	98	5	-
Flex Ore	-	kt	39,839	2,521	2,701	1,480	1,814	3,872	1,189	1,186	1,660	825	1,902	3,597	4,372	2,045	1,071	584	17	-
Total RoM	**-**	**kt**	**107,302**	**5,080**	**5,480**	**4,369**	**3,812**	**7,199**	**3,065**	**3,541**	**3,787**	**3,111**	**4,942**	**7,227**	**7,763**	**5,832**	**3,767**	**2,523**	**368**	**-**
Total Mine Movement	**-**	**kt**	**665,246**	**27,092**	**27,437**	**27,688**	**33,401**	**34,872**	**33,223**	**33,459**	**33,360**	**33,055**	**34,202**	**34,994**	**34,072**	**34,077**	**28,877**	**18,876**	**1,406**	**-**
Material Rehandling	-	kt	29,109	1,248	677	1,842	1,348	437	2,045	1,569	1,323	2,013	498	306	381	1,124	1,582	2,587	2,877	-
Total Movement		**kt**	**694,355**	28,339	28,114	29,530	34,749	35,309	35,267	35,028	34,683	35,067	34,700	35,301	34,453	35,201	30,459	21,462	4,283	-
Flotation Plant Feed																				
Ore Feed to Flotation		kt	46,981	2,562	2,555	2,555	2,555	2,562	2,555	2,555	2,555	2,562	2,555	2,555	2,555	2,562	2,555	2,555	1,346	-
Total Flotation Feed		**kt**	**46,981**	2,562	2,555	2,555	2,555	2,562	2,555	2,555	2,555	2,562	2,555	2,555	2,555	2,562	2,555	2,555	1,346	-
Flotation Plant Feed Rate																				
Ore Feed to Flotation		tpd	6,770	7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	3,687	-
Total Flotation Feed Rate		**tpd**	**6,770**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**	**3,687**	**-**
CIL Feed																				
Main/SE Saprolite Feed to CIL	-	kt	9,132	384	383	383	383	384	383	383	383	384	383	383	383	262	-	-	-	-
Main/SE Fresh Ore Feed to CIL	-	kt	43,882	2,178	2,172	2,172	2,172	2,178	2,172	2,172	2,172	2,178	2,172	2,172	2,172	2,300	2,555	2,555	1,899	-
SH Saprolite Feed to CIL	-	kt	0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
SH Fresh Ore Feed to CIL	-	kt	7,307	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total CIL Feed		**kt**	**60,322**	2,562	2,555	2,555	2,555	2,562	2,555	2,555	2,555	2,562	2,555	2,555	2,555	2,562	2,555	2,555	1,899	-
CIL Feed Rate																				
Main/SE Saprolite Feed to CIL	-	tpd	1,191	1,050	1,050	1,050	1,050	1,050	1,050	1,050	1,050	1,050	1,050	1,050	1,050	717	-	-	-	-
Main/SE Fresh Ore Feed to CIL	-	tpd	5,006	5,950	5,950	5,950	5,950	5,950	5,950	5,950	5,950	5,950	5,950	5,950	5,950	6,283	7,000	7,000	5,203	-
SH Saprolite Feed to CIL	-	tpd	0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
SH Fresh Ore Feed to CIL	-	tpd	2,859	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total CIL Feed Rate		**tpd**	**6,881**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**	**7,000**	**5,203**	**-**
Cumulative Process Head Grade																				
Cumulative CIL Feed Au	**-**	g/t	**2.16**	2.25	2.16	2.11	2.09	2.09	2.11	2.12	2.10	2.11	2.18	2.22	2.21	2.24	2.22	2.19	2.16	2.16
Cumulative Flotation Feed Au	**-**	g/t	**1.11**	1.09	1.15	1.12	1.12	1.20	1.17	1.14	1.13	1.11	1.10	1.14	1.17	1.18	1.16	1.13	1.11	1.11
Cumulative Au Grade	**-**	g/t	**1.70**	1.90	1.83	1.77	1.74	1.75	1.74	1.73	1.71	1.70	1.73	1.77	1.77	1.78	1.76	1.73	1.70	1.70
Cumulative Flotation Feed Cu	**-**	**%**	**0.17%**	0.19%	0.20%	0.20%	0.20%	0.20%	0.20%	0.19%	0.19%	0.18%	0.18%	0.18%	0.18%	0.18%	0.18%	0.17%	0.17%	0.17%
				1.32	**1.24**	**1.29**	**1.41**	**1.37**	**1.49**	**1.56**	**1.59**	**1.64**	**1.70**	**1.68**	**1.60**	**1.60**	**1.61**	**1.63**	**1.66**	
Production																				
Gold in Dore	-	koz	4,337	146	130	145	164	199	199	199	162	186	286	270	195	237	157	136	76	-
Gold in Concentrate	-	koz	1,106	70	76	54	63	93	51	47	54	48	56	84	83	65	47	38	16	-
Total Recovered Gold	**0.883573268**	**koz**	**5,443**	**215**	**206**	**199**	**227**	**293**	**250**	**246**	**216**	**234**	**342**	**354**	**278**	**302**	**204**	**174**	**92**	**-**
Cumulative Recovered Gold				**1,827**	**2,033**	**2,232**	**2,459**	**2,751**	**3,001**	**3,248**	**3,464**	**3,697**	**4,039**	**4,393**	**4,671**	**4,973**	**5,177**	**5,351**	**5,443**	**5,443**
Cumulative avg. Annual Gold Produced		**kozs/y**	**233**	**203**	**203**	**203**	**205**	**212**	**214**	**217**	**216**	**217**	**224**	**231**	**234**	**237**	**235**	**233**	**227**	
Copper Concentrate	-	kt	311	19	21	23	17	25	15	10	15	15	13	16	16	17	20	10	4	-
Copper in Concentrate		kt	67.2	**4.1**	**4.6**	**5.0**	**3.6**	**5.6**	**3.1**	**2.1**	**3.1**	**3.3**	**2.7**	**3.5**	**3.5**	**3.6**	**4.4**	**2.1**	**0.8**	**-**
Copper in Concentrate	-	klb	148,141	9,089	10,163	11,026	7,944	12,238	6,926	4,571	6,919	7,243	6,030	7,693	7,619	7,889	9,770	4,620	1,841	-
Payable Gold																				
Payable Gold in Dore	-	koz	4,335	145	130	145	164	199	199	199	162	186	286	270	195	237	157	136	76	-
Payable Gold in Concentrate	-	koz	1,073	68	73	53	61	91	50	46	52	46	54	82	81	63	46	37	15	-
Total Payable Gold	**-**	**koz**	**5,407**	**213**	**203**	**198**	**225**	**290**	**248**	**245**	**214**	**232**	**340**	**352**	**275**	**300**	**203**	**173**	**91**	**-**
Sold at Spot		**koz**	**4,867**	192	183	178	202	261	223	220	193	209	306	316	248	270	182	155	82	-
Sold to SLW		**koz**	**541**	21	20	20	22	29	25	24	21	23	34	35	28	30	20	17	9	-
Payable Silver																				
Payable Silver in Dore	-	koz	1,292	59	62	31	41	69	62	39	39	24	41	56	61	47	32	20	50	-
Paynle Silver in Concentrate	-	koz	1,230	103	117	66	36	86	62	34	45	33	50	68	79	68	66	37	22	-
Total Payable Silver	**-**	**koz**	**2,522**	163	179	97	77	155	124	74	83	58	91	124	140	115	98	56	72	-
Sold at Spot		**koz**	**1,261**	81	90	49	39	77	62	37	42	29	46	62	70	57	49	28	36	-
Sold to SLW		**koz**	**1,261**	81	90	49	39	77	62	37	42	29	46	62	70	57	49	28	36	-
Payable Copper	**-**	**klb**	**141,295**	**8,670**	**9,696**	**10,521**	**7,576**	**11,680**	**6,604**	**4,357**	**6,597**	**6,907**	**5,749**	**7,336**	**7,265**	**7,523**	**9,321**	**4,404**	**1,754**	**-**
Payable Copper	-	t	64,090																	
Tailings																				
Flotation Tailings	-	kt	46,670	2,543	2,534	2,532	2,538	2,537	2,540	2,545	2,540	2,547	2,542	2,539	2,539	2,545	2,535	2,545	1,342	-
CIL Tailings	-	kt	60,322	2,562	2,555	2,555	2,555	2,562	2,555	2,555	2,555	2,562	2,555	2,555	2,555	2,562	2,555	2,555	1,899	-
Total Tailings	**-**	**kt**	**106,992**	**5,105**	**5,089**	**5,087**	**5,093**	**5,099**	**5,095**	**5,100**	**5,095**	**5,109**	**5,097**	**5,094**	**5,094**	**5,107**	**5,090**	**5,100**	**3,241**	**-**

	value / factor	units / sensit.	Total or Avg.	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048
				9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25
CASH FLOW SCHEDULE																				
Price Schedule																				
Gold - Sold at Spot		US$/oz	-	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500
Gold - Sold to SLW		US$/oz	-	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400
Silver - Sold at Spot		US$/oz	-	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20
Silver - Sold to SLW		US$/oz	-	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Copper		US$/lb	-	3.13	3.13	3.13	3.13	3.13	3.13	3.13	3.13	3.13	3.13	3.13	3.13	3.13	3.13	3.13	3.13	3.13
Net Revenue Before By-Product Credits																				
Gold	94%	$000s	7,516,386	296,434	282,415	274,864	312,195	402,662	345,140	340,253	297,767	322,816	472,361	488,602	382,607	416,980	281,744	239,793	127,010	0
Silver	0%	$000s	30,413	1,961	2,161	1,174	929	1,869	1,492	887	1,005	695	1,100	1,491	1,686	1,385	1,178	680	865	0
Copper	6%	$000s	441,660	27,100	30,306	32,885	23,681	36,509	20,642	13,620	20,621	21,589	17,971	22,932	22,710	23,516	29,135	13,766	5,484	0
Total Net Revenue Before By-Product Credits	-	**$000s**	**7,988,460**	**325,495**	**314,882**	**308,923**	**336,805**	**441,040**	**367,274**	**354,760**	**319,393**	**345,100**	**491,431**	**513,025**	**407,002**	**441,882**	**312,056**	**254,239**	**133,359**	**0**
Net Revenue																				
Gold	-	$000s	7,516,386	296,434	282,415	274,864	312,195	402,662	345,140	340,253	297,767	322,816	472,361	488,602	382,607	416,980	281,744	239,793	127,010	0
Silver	-	$000s	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Copper	-	$000s	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total Net Revenue	-	**$000s**	**7,516,386**	**296,434**	**282,415**	**274,864**	**312,195**	**402,662**	**345,140**	**340,253**	**297,767**	**322,816**	**472,361**	**488,602**	**382,607**	**416,980**	**281,744**	**239,793**	**127,010**	**0**
Direct Cash Costs	($4.14)																			
Mining Cost (OP & UG)	($17.16)	$000s	(1,840,872)	(54,770)	(56,342)	(95,740)	(115,636)	(113,661)	(120,020)	(119,404)	(115,094)	(117,300)	(113,318)	(112,451)	(108,221)	(109,818)	(85,439)	(38,854)	(8,895)	0
Processing Cost	($14.52)	$000s	(1,557,511)	(73,504)	(73,321)	(73,334)	(73,260)	(73,550)	(73,248)	(73,156)	(73,232)	(73,466)	(73,262)	(73,165)	(73,114)	(73,437)	(74,261)	(73,770)	(49,688)	0
Site G&A Cost	($3.36)	$000s	(360,096)	(16,626)	(16,630)	(16,716)	(16,447)	(16,478)	(16,090)	(14,186)	(14,170)	(14,185)	(14,117)	(13,881)	(13,800)	(13,689)	(13,332)	(12,853)	(13,625)	0
Freight Cost	($0.43)	$000s	(46,297)	(2,837)	(3,159)	(3,416)	(2,490)	(3,774)	(2,178)	(1,446)	(2,176)	(2,276)	(1,902)	(2,413)	(2,391)	(2,473)	(3,042)	(1,462)	(583)	0
Treatment Charges	($0.17)	$000s	(18,634)	(1,142)	(1,271)	(1,375)	(1,002)	(1,519)	(877)	(582)	(876)	(916)	(765)	(971)	(962)	(995)	(1,224)	(588)	(235)	0
Refining Charges	($0.17)	$000s	(18,019)	(1,011)	(1,090)	(1,037)	(902)	(1,339)	(838)	(674)	(807)	(816)	(884)	(1,097)	(1,021)	(992)	(945)	(576)	(257)	0
Predicted Penalties	($0.01)	$000s	(1,553)	(95)	(106)	(115)	(84)	(127)	(73)	(49)	(73)	(76)	(64)	(81)	(80)	(83)	(102)	(49)	(20)	0
By-Product Credit	$4.40	$000s	472,073	29,061	32,467	34,059	24,611	38,378	22,134	14,507	21,627	22,284	19,070	24,424	24,396	24,901	30,313	14,446	6,349	0
Total Direct Cash Costs	1	**$000s**	**(3,370,909)**	**(120,925)**	**(119,453)**	**(157,673)**	**(185,210)**	**(172,070)**	**(191,191)**	**(194,990)**	**(184,801)**	**(186,752)**	**(185,242)**	**(179,636)**	**(175,193)**	**(176,587)**	**(148,033)**	**(113,706)**	**(66,954)**	**0**
Direct Cash Cost	**-**	**US$/oz-Au**	**623**	**567**	**588**	**797**	**825**	**594**	**770**	**797**	**863**	**804**	**545**	**511**	**636**	**589**	**730**	**659**	**733**	**0**
Indirect Cash Costs																				
Royalties	($5.81)	$000s	(623,952)	**(25,908)**	**(24,826)**	**(24,274)**	**(25,078)**	**(32,427)**	**(27,742)**	**(27,230)**	**(23,921)**	**(25,936)**	**(37,817)**	**(39,104)**	**(30,659)**	**(33,460)**	**(22,797)**	**(19,236)**	**(10,210)**	**0**
Freight Insurance Cost	-	$000s	(17,211)	**(709)**	**(694)**	**(613)**	**(687)**	**(933)**	**(797)**	**(722)**	**(653)**	**(664)**	**(969)**	**(1,035)**	**(857)**	**(889)**	**(622)**	**(493)**	**(359)**	**0**
Total Indirect Cash Costs		**$000s**	**(641,163)**	**(26,618)**	**(25,520)**	**(24,887)**	**(25,765)**	**(33,360)**	**(28,539)**	**(27,952)**	**(24,574)**	**(26,601)**	**(38,786)**	**(40,139)**	**(31,516)**	**(34,349)**	**(23,419)**	**(19,729)**	**(10,569)**	**0**
Indirect Cash Cost		**US$/oz-Au**	**119**	**125**	**126**	**126**	**115**	**115**	**115**	**114**	**115**	**115**	**114**	**114**	**114**	**115**	**116**	**114**	**116**	**0**
Sustaining Cash Cost		**US$/oz-Au**	**175**	**384**	**412**	**266**	**315**	**118**	**28**	**62**	**138**	**189**	**14**	**191**	**21**	**48**	**36**	**9**	**243**	**0**
AISC Cost		**US$/oz-Au**	**916**	**1,076**	**1,126**	**1,190**	**1,254**	**827**	**913**	**973**	**1,115**	**1,107**	**673**	**817**	**772**	**751**	**882**	**783**	**1,092**	**0**
Cumulative AISC				***903***	***925***	***949***	***977***	***961***	***957***	***958***	***968***	***977***	***951***	***940***	***930***	***920***	***918***	***914***	***917***	
Total Operating Expense	-	**$000s**	**(4,012,072)**	**(147,543)**	**(144,973)**	**(182,560)**	**(210,976)**	**(205,429)**	**(219,729)**	**(222,942)**	**(209,375)**	**(213,352)**	**(224,028)**	**(219,775)**	**(206,709)**	**(210,936)**	**(171,452)**	**(133,435)**	**(77,523)**	**0**
Operating Margin	-	**$000s**	**3,504,315**	**148,891**	**137,442**	**92,304**	**101,219**	**197,232**	**125,411**	**117,311**	**88,392**	**109,464**	**248,333**	**268,827**	**175,897**	**206,044**	**110,292**	**106,358**	**49,487**	**0**
Cash Available for Debt Service																				
Operating Margin		$000s	3,504,315	148,891	137,442	92,304	101,219	197,232	125,411	117,311	88,392	109,464	248,333	268,827	175,897	206,044	110,292	106,358	49,487	0
Project Capital	**(354,760)**	$000s	(1,298,570)	(81,934)	(83,772)	(52,675)	(70,703)	(34,250)	(7,007)	(15,272)	(29,551)	(43,805)	(4,820)	(67,242)	(5,912)	(14,260)	(7,335)	(1,562)	(22,216)	0
Income Tax	-	$000s	(649,572)	(25,427)	(28,239)	(12,367)	(11,435)	(39,770)	(22,719)	(24,399)	(17,111)	(25,046)	(68,473)	(71,007)	(43,689)	(53,651)	(27,113)	(26,129)	(11,769)	0
Working Capital	-	$000s	0	(187)	(279)	(7,899)	(3,911)	331	(1,134)	522	850	(491)	851	240	872	(362)	4,782	9,511	10,654	14,442
Cash Flow	**(354,760)**	**$000s**	**1,556,172**	**41,343**	**25,152**	**19,362**	**15,170**	**123,543**	**94,551**	**78,162**	**42,581**	**40,122**	**175,891**	**130,818**	**127,168**	**137,772**	**80,626**	**88,178**	**26,155**	**14,442**
GPA Installment Payments	138,000	$000s	138,000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Loan Repayment	-	$000s	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Interest Expense	-	$000s	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Free Cash Flow	**(216,760)**	**$000s**	**1,694,172**	**41,343**	**25,152**	**19,362**	**15,170**	**123,543**	**94,551**	**78,162**	**42,581**	**40,122**	**175,891**	**130,818**	**127,168**	**137,772**	**80,626**	**88,178**	**26,155**	**14,442**
ECONOMICS METRICS																				
Cumulative FCF			**1.15**	**474,479**	**499,631**	**518,993**	**534,163**	**657,706**	**752,257**	**830,420**	**873,000**	**913,122**	**1,089,014**	**1,219,831**	**1,346,999**	**1,484,771**	**1,565,397**	**1,653,575**	**1,679,730**	**1,694,172**
After-Tax IRR			**46.08%**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
NPV Period				**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**	**20**	**21**	**22**	**23**	**24**	**25**	**26**	**27**	**28**
After-Tax Present Value	0%	$000s	**1,694,172**	**41,343**	**25,152**	**19,362**	**15,170**	**123,543**	**94,551**	**78,162**	**42,581**	**40,122**	**175,891**	**130,818**	**127,168**	**137,772**	**80,626**	**88,178**	**26,155**	**14,442**
After-Tax Present Value	3%	$000s	**1,058,018**	**29,867**	**17,641**	**13,184**	**10,029**	**79,298**	**58,921**	**47,289**	**25,012**	**22,881**	**97,387**	**70,321**	**66,368**	**69,808**	**39,663**	**42,114**	**12,128**	**6,501**
After-Tax Present Value	5%	$000s	**794,034**	**24,172**	**14,006**	**10,268**	**7,662**	**59,427**	**43,315**	**34,102**	**17,693**	**15,878**	**66,292**	**46,956**	**43,472**	**44,854**	**25,000**	**26,039**	**7,356**	**3,868**
After-Tax Present Value	8%	$000s	**535,423**	**17,731**	**9,988**	**7,119**	**5,165**	**38,946**	**27,599**	**21,125**	**10,656**	**9,297**	**37,737**	**25,988**	**23,391**	**23,465**	**12,715**	**12,876**	**3,536**	**1,808**
After-Tax Present Value	10%	$000s	**420,676**	**14,491**	**8,014**	**5,608**	**3,995**	**29,575**	**20,577**	**15,464**	**7,659**	**6,560**	**26,145**	**17,677**	**15,622**	**15,386**	**8,186**	**8,138**	**2,195**	**1,102**
Cumulative After-Tax Present Value	5%	$000s	-	**327,847**	**341,852**	**352,120**	**359,782**	**419,209**	**462,524**	**496,626**	**514,319**	**530,197**	**596,488**	**643,444**	**686,916**	**731,771**	**756,770**	**782,810**	**790,166**	**794,034**
Cumulative After-Tax Present Value	8%	$000s	-	**264,014**	**274,002**	**281,121**	**286,286**	**325,232**	**352,831**	**373,956**	**384,611**	**393,908**	**431,645**	**457,633**	**481,024**	**504,489**	**517,204**	**530,079**	**533,615**	**535,423**
Cumulative After-Tax Present Value	10%	$000s	-	**228,772**	**236,786**	**242,395**	**246,390**	**275,965**	**296,542**	**312,006**	**319,665**	**326,225**	**352,370**	**370,047**	**385,669**	**401,056**	**409,241**	**417,380**	**419,574**	**420,676**
			0.92																	
Pre-Tax IRR			**58.53%**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
Pre-Tax Present Value	0%		**2,343,744**	**66,770**	**53,391**	**31,729**	**26,605**	**163,313**	**117,270**	**102,561**	**59,691**	**65,168**	**244,364**	**201,824**	**170,857**	**191,422**	**107,739**	**114,307**	**37,924**	**14,442**
Pre-Tax Present Value	5%		**1,108,302**	**39,039**	**29,730**	**16,827**	**13,437**	**78,556**	**53,723**	**44,747**	**24,803**	**25,789**	**92,098**	**72,443**	**58,407**	**62,322**	**33,407**	**33,755**	**10,666**	**3,868**
Pre-Tax Present Value	8%		**754,482**	**28,637**	**21,202**	**11,667**	**9,058**	**51,483**	**34,230**	**27,719**	**14,938**	**15,100**	**52,428**	**40,094**	**31,428**	**32,602**	**16,990**	**16,691**	**5,127**	**1,808**
EBTIDA	-		**3,504,315**	**148,891**	**137,442**	**92,304**	**101,219**	**197,232**	**125,411**	**117,311**	**88,392**	**109,464**	**248,333**	**268,827**	**175,897**	**206,044**	**110,292**	**106,358**	**49,487**	**-**
Cumulative Pre-Tax FCF				**641,135**	**694,526**	**726,255**	**752,861**	**916,174**	**1,033,444**	**1,136,005**	**1,195,696**	**1,260,864**	**1,505,228**	**1,707,052**	**1,877,909**	**2,069,332**	**2,177,071**	**2,291,378**	**2,329,303**	**2,343,744**
Begin Equity		$000s		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
GPA Installment Payments		$000s	**138,000**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Debt		$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Construction Cost or Cover of Losses		$000s	**(402,597)**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Additional Equity		$000s	**216,760**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
End		$000s		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Initial Equity		$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Additional Equity		$000s	**(264,597)**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Dividends		$000s	**1,958,769**	**41,343**	**25,152**	**19,362**	**15,170**	**123,543**	**94,551**	**78,162**	**42,581**	**40,122**	**175,891**	**130,818**	**127,168**	**137,772**	**80,626**	**88,178**	**26,155**	**14,442**
Net Cashflow		$000s	**1,694,172**	**41,343**	**25,152**	**19,362**	**15,170**	**123,543**	**94,551**	**78,162**	**42,581**	**40,122**	**175,891**	**130,818**	**127,168**	**137,772**	**80,626**	**88,178**	**26,155**	**14,442**
Return on Equity			**46.08%**																	

	value / factor	units / sensit.	Total or Avg.	2032 9	2033 10	2034 11	2035 12	2036 13	2037 14	2038 15	2039 16	2040 17	2041 18	2042 19	2043 20	2044 21	2045 22	2046 23	2047 24	2048 M 25
COPPER CONCENTRATE FREIGHT																				
Freight & Marketing	$/t-conc		0.0%																	
Road Transport	$12.15	$000s	(4,101)	(251)	(280)	(303)	(221)	(334)	(193)	(128)	(193)	(202)	(168)	(214)	(212)	(219)	(269)	(129)	(52)	0
Rail Freight	$0.00	$000s	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Handling Charge	$0.00	$000s	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Ocean Freight	$100.00	$000s	(33,756)	(2,069)	(2,303)	(2,491)	(1,815)	(2,752)	(1,588)	(1,054)	(1,587)	(1,659)	(1,387)	(1,760)	(1,743)	(1,803)	(2,218)	(1,066)	(425)	0
Container Discharge Fee	$25.00	$000s	(8,439)	(517)	(576)	(623)	(454)	(688)	(397)	(264)	(397)	(415)	(347)	(440)	(436)	(451)	(554)	(266)	(106)	0
Subtotal		**$000s**	**(46,297)**	**(2,837)**	**(3,159)**	**(3,416)**	**(2,490)**	**(3,774)**	**(2,178)**	**(1,446)**	**(2,176)**	**(2,276)**	**(1,902)**	**(2,413)**	**(2,391)**	**(2,473)**	**(3,042)**	**(1,462)**	**(583)**	**0**
ROYALTIES																				
Royalties																				
Guyana Au Royalty	8.0%	$000s	(595,750)	(23,398)	(22,257)	(21,739)	(24,686)	(31,791)	(27,354)	(26,983)	(23,567)	(25,593)	(37,498)	(38,688)	(30,230)	(33,042)	(22,314)	(19,001)	(10,072)	0
Guyana Ag Royalty	8.0%	$000s	(2,018)	(130)	(144)	(75)	(61)	(124)	(99)	(56)	(65)	(42)	(67)	(95)	(111)	(88)	(75)	(42)	(62)	0
Guyana Cu Royalty	1.5%	$000s	(6,185)	(380)	(424)	(461)	(332)	(512)	(289)	(191)	(289)	(302)	(252)	(321)	(318)	(329)	(408)	(193)	(77)	0
One Time Royalty to Surface Owner	-	$000s	(20,000)	(2,000)	(2,000)	(2,000)	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total Royalties		**$000s**	**(623,952)**	**(25,908)**	**(24,826)**	**(24,274)**	**(25,078)**	**(32,427)**	**(27,742)**	**(27,230)**	**(23,921)**	**(25,936)**	**(37,817)**	**(39,104)**	**(30,659)**	**(33,460)**	**(22,797)**	**(19,236)**	**(10,210)**	**0**
PROJECT CAPITAL			LoM																	
Project Capital																				
Pre-Stripping Capex - Mining Costs		$000s	31,774	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Pre-Stripping Capex - G&A Costs		$000s	9,242	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Mine (OP & UG)		$000s	694,913	64,712	83,772	52,675	70,703	34,250	7,007	15,272	29,551	43,805	4,820	67,242	5,912	14,260	7,335	1,562	0	0
Process Plant		$000s	198,220	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Water and Tailings Management		$000s	45,299	14,976	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Infrastructure		$000s	69,411	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Power Supply		$000s	3,228	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Owner's		$000s	66,402	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	22,216	0
Indirect Costs		$000s	112,832	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Risk and Contingency		$000s	67,249	2,246	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total Capital	**1**	**354,760**	**1,298,570**	**81,934**	**83,772**	**52,675**	**70,703**	**34,250**	**7,007**	**15,272**	**29,551**	**43,805**	**4,820**	**67,242**	**5,912**	**14,260**	**7,335**	**1,562**	**22,216**	**0**
Cum. Capital				**838,187**	**921,959**	**974,634**	**1,045,338**	**1,079,588**	**1,086,595**	**1,101,867**	**1,131,418**	**1,175,223**	**1,180,043**	**1,247,285**	**1,253,197**	**1,267,457**	**1,274,792**	**1,276,354**	**1,298,570**	**1,298,570**
Initial Capital			**354,760**																	
Sustaining Capital			**943,811**	**81,934**	**83,772**	**52,675**	**70,703**	**34,250**	**7,007**	**15,272**	**29,551**	**43,805**	**4,820**	**67,242**	**5,912**	**14,260**	**7,335**	**1,562**	**22,216**	**0**
Working Capital																				
Beginning Balance	-	$000s	751,696	28,793	28,980	29,259	37,158	41,069	40,738	41,872	41,349	40,499	40,990	40,140	39,899	39,027	39,389	34,607	25,095	14,442
Ending Balance	**20%**	$000s	751,696	28,980	29,259	37,158	41,069	40,738	41,872	41,349	40,499	40,990	40,140	39,899	39,027	39,389	34,607	25,095	14,442	0
Change	-	**$000s**	**0**	**(187)**	**(279)**	**(7,899)**	**(3,911)**	**331**	**(1,134)**	**522**	**850**	**(491)**	**851**	**240**	**872**	**(362)**	**4,782**	**9,511**	**10,654**	**14,442**
INCOME TAX			**LoM**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**	**20**	**21**	**22**	**23**	**24**	**25**
Income Tax																				
Net Revenue	-	$000s	7,516,386	296,434	282,415	274,864	312,195	402,662	345,140	340,253	297,767	322,816	472,361	488,602	382,607	416,980	281,744	239,793	127,010	0
Operating Expenses	-	$000s	(4,012,072)	(147,543)	(144,973)	(182,560)	(210,976)	(205,429)	(219,729)	(222,942)	(209,375)	(213,352)	(224,028)	(219,775)	(206,709)	(210,936)	(171,452)	(133,435)	(77,523)	0
Operating Profit	-	**$000s**	**3,504,315**	**148,891**	**137,442**	**92,304**	**101,219**	**197,232**	**125,411**	**117,311**	**88,392**	**109,464**	**248,333**	**268,827**	**175,897**	**206,044**	**110,292**	**106,358**	**49,487**	**0**
Interest Expense	-	$000s	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Depreciation	-	$000s	(1,298,570)	(64,134)	(43,311)	(51,079)	(63,102)	(64,667)	(49,682)	(35,982)	(31,357)	(25,977)	(20,091)	(32,138)	(30,266)	(27,208)	(19,914)	(19,262)	(10,257)	(9,075)
Net Income	-	**$000s**	**2,205,744**	**84,757**	**94,131**	**41,225**	**38,117**	**132,566**	**75,729**	**81,329**	**57,035**	**83,487**	**228,242**	**236,689**	**145,631**	**178,836**	**90,378**	**87,096**	**39,230**	**(9,075)**
Loss Carry Forward			0																	
Additions	-	**$000s**	**25,407**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**9,075**
Opening Balance	-	$000s		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	9,075
Losses Used	-	$000s	65,911	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Closing Balance	-	$000s	509,794	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	9,075
Loss Carry Forward	-	$000s	(40,504)	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	9,075
Taxable Income	-	**$000s**	**2,165,241**	**84,757**	**94,131**	**41,225**	**38,117**	**132,566**	**75,729**	**81,329**	**57,035**	**83,487**	**228,242**	**236,689**	**145,631**	**178,836**	**90,378**	**87,096**	**39,230**	**0**
Effective Income Tax	**30%**	**$000s**	**649,572**	**25,427**	**28,239**	**12,367**	**11,435**	**39,770**	**22,719**	**24,399**	**17,111**	**25,046**	**68,473**	**71,007**	**43,689**	**53,651**	**27,113**	**26,129**	**11,769**	**0**