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NI 43-101 TECHNICAL REPORT ON THE PRE-FEASIBILITY STUDY FOR THE FENELON MINE PROPERTY

Project Location

Latitude 50°01'00" North and Longitude 78°37'30"
Province of Québec, Canada

Prepared for

Wallbridge Mining Company Ltd
129 Fielding Road
Lively, Ontario
Canada P3Y 1L7

Prepared by:

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Effective Date: February 2, 2017
Original Date: March 3, 2017

SIGNATURE PAGE – INNOVEXPLO INC.

NI 43-101 TECHNICAL REPORT ON THE PRE-FEASIBILITY STUDY FOR THE FENELON MINE PROPERTY

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Lively, Ontario,
Canada P3Y 1L7

(Original signed and sealed)
Pierre-Luc Richard, P. Geo.
InnovExplo Inc.
Val-d'Or (Québec)

Signed at Val-d'Or on March 3, 2017

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Signed at Val-d'Or on March 3, 2017

(Original signed and sealed)
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Signed at Longueuil on March 3, 2017

SIGNATURE PAGE – WSP CANADA INC.

NI 43-101 TECHNICAL REPORT ON THE PRE-FEASIBILITY STUDY FOR THE FENELON MINE PROPERTY

Project Location

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Province of Québec, Canada

Prepared for

Wallbridge Mining Company Ltd
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Marie Claude Dion St-Pierre, P. Eng.
WSP Canada Inc.
Québec (Québec)

Signed at the city of Québec on March 3, 2017

SIGNATURE PAGE – SNC-LAVALIN INC.

NI 43-101 TECHNICAL REPORT ON THE PRE-FEASIBILITY STUDY FOR THE FENELON MINE PROPERTY

Project Location

Latitude 50°01'00" North and Longitude 78°37'30"
Province of Québec, Canada

Prepared for

Wallbridge Mining Company Ltd.
129 Fielding Road
Lively, Ontario,
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(Original signed and sealed)
George Darling, P. Eng.
SNC-Lavalin Inc.
Sudbury (Ontario)

Signed at Sudbury on March 3, 2017

CERTIFICATE OF AUTHOR – PIERRE-LUC RICHARD

I, Pierre-Luc Richard, M.Sc., P.Geo. (OGQ licence No. 1119; APGO licence No. 1714; APEGBC licence No. 43255; NAPEG licence No. L2465; MAusIMM), do hereby certify that:

1. I am employed as a geologist by and carried out this assignment for: InnovExplo Inc. – Consulting Firm in Mines and Exploration, 560, 3e Avenue, Val-d'Or, Québec, Canada, J9P 1S4.
2. I graduated with a Bachelor's degree in geology from the Université du Québec à Montreal (Montreal, Québec) in 2004. In addition, I obtained an M.Sc. from the Université du Québec à Chicoutimi (Chicoutimi, Québec) in 2012.
3. I am a member in good standing of the Ordre des Géologues du Québec (OGQ licence No. 1119), of the Association of Professional Geoscientists of Ontario (APGO licence No. 1714), of the Association of Professional Engineers and Geoscientists of British Columbia (APEGBC licence No. 43255), of the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (NAPEG licence No. L2465), and of the Australian AusIMM Minerals Institute.
4. I have worked in the mining industry for more than 10 years. My exploration expertise has been acquired with Richmont Mines Inc., the Ministry of Natural Resources of Québec (Geology Branch), and numerous exploration companies through InnovExplo. My mining expertise was acquired at the Beaufor mine and several other producers through InnovExplo. I managed numerous technical reports, resource estimates and audits. I have been a geological consultant for InnovExplo Inc. since February 2007 and I currently hold the Director of Geology position.
5. I have read the definition of "qualified person" set out in Regulation 43-101 / 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.
6. I am co-author of and share the responsibility for sections 1, 14 and 25 to 27 of the technical report entitled "NI 43-101 TECHNICAL REPORT ON THE PRE-FEASIBILITY STUDY FOR THE FENELON MINE PROPERTY", prepared for Wallbridge Mining Company Ltd. The effective date of the report is February 2, 2017, and the signature date is March 3, 2017.
7. I did not visit the property that is the subject of the Technical Report.
8. I have had prior involvement with the project as I was co-author of the technical report entitled "TECHNICAL REPORT AND MINERAL RESOURCE ESTIMATE FOR THE FENELON MINE PROJECT (according to National Instrument 43-101 and Form 43-010F)", effective date of July 5, 2016, and signature date of July 29, 2016 (revised on August 17, 2016), prepared for Wallbridge Mining Company Ltd.
9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Report, the omission of which would make the Technical Report misleading.
10. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
11. I have read NI 43-101 Respecting Standards of Disclosure for Mineral projects and Form 43-101F1, and the items for which I am a qualified person in this Technical Report have been prepared in accordance with that regulation and form.

Dated this 3rd day of March 2017, in the city of Val-d'Or (Québec)

(Original signed and sealed)

Pierre-Luc Richard, P.Geo.
InnovExplo Inc.
pierreluc.richard@innovexplo.com

CERTIFICATE OF AUTHOR – BRUNO TURCOTTE

I, Bruno Turcotte, P.Geo. (APGO licence No. 2136; OGQ licence No. 453), do hereby certify that:

1. I am employed as a geologist by and carried out this assignment for: InnovExplo Inc. – Consulting Firm in Mines and Exploration, 560, 3e Avenue, Val-d'Or, Québec, Canada, J9P 1S4.
2. I graduated with a Bachelor of Geology degree from Université Laval in the city of Québec in 1995. In addition, I obtained a Master's degree in Earth Sciences from Université Laval in the city of Québec in 1999.
3. I am a member of the Ordre des Géologues du Québec (OGQ licence No. 453) and of the Association of Professional Geoscientists of Ontario (APGO licence No. 2136).
4. I have worked as a geologist for a total of 21 years since graduating from university. I acquired my exploration expertise with Noranda Exploration Inc., Breakwater Resources Ltd, South-Malartic Exploration Inc. and Richmont Mines Inc. I acquired my mining expertise on the Croinor Preproduction Project and at the Beaufor mine. I have been a geological consultant for InnovExplo Inc. since March 2007.
5. I have read the definition of "qualified person" set out in Regulation 43-101 / 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.
6. I am the author of and responsible for sections 2 to 11, 23 and 24, and co-author of and share the responsibility for sections 1, 25, 26 and 27, of the technical report entitled "NI 43-101 TECHNICAL REPORT ON THE PRE-FEASIBILITY STUDY FOR THE FENELON MINE PROPERTY", prepared for Wallbridge Mining Company Ltd. The effective date of the report is February 2, 2017, and the signature date is March 3, 2017.
7. I have had prior involvement with the project as I was co-author of the technical report entitled "TECHNICAL REPORT AND MINERAL RESOURCE ESTIMATE FOR THE FENELON MINE PROJECT (according to National Instrument 43-101 and Form 43-010F)", effective date of July 5, 2016, and signature date of July 29, 2016 (revised on August 17, 2016), prepared for Wallbridge Mining Company Ltd.
8. I have not conducted a site visit of the property that is the subject of the Technical Report.
9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission of which would make the Technical Report misleading.
10. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
11. I have read NI 43-101 Respecting Standards of Disclosure for Mineral projects and Form 43-101F1, and the Technical Report has been prepared in accordance with that instrument and form.

Dated this 3rd day of March 2017, in the city of Val-d'Or (Québec)

(Original signed and sealed)

Bruno Turcotte, P.Geo.
InnovExplo Inc.
bruno.turcotte@innovexplo.com

CERTIFICATE OF AUTHOR – CATHERINE JALBERT

I, Catherine Jalbert, P.Geo., B.Sc. (OGQ licence No. 1412; NAPEG licence No. L3534) do hereby certify that:

1. I am employed as a geologist by and carried this assignment for InnovExplo Inc. – Consulting Firm in Mines and Exploration, 560, 3e Avenue, Val-d'Or, Québec, Canada, J9P 1S4.
2. I graduated with a Bachelor's degree in geology from the Université Laval in Québec City, Québec in 2009.
3. I am a member in good standing of the Ordre des Géologues du Québec (OGQ licence No. 1412) and of the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (NAPEG licence No. L3534).
4. I have worked in the mining industry for more than 7 years. My exploration expertise has been acquired with numerous exploration companies through InnovExplo. I have been a geological consultant for InnovExplo Inc. since May 2009.
5. I have read the definition of "qualified person" set out in Regulation 43-101 / 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.
6. I am author of and responsible for section 12, and I am co-author of and share the responsibility for sections 1, 14, 25, 26 and 27, of the technical report entitled "NI 43-101 TECHNICAL REPORT ON THE PRE-FEASIBILITY STUDY FOR THE FENELON MINE PROPERTY", prepared for Wallbridge Mining Company Ltd. The effective date of the report is February 2, 2017, and the signature date is March 3, 2017.
7. I visited the property that is the subject of the Technical Report on May 31 and June 1, 2016.
8. I have had prior involvement with the project as I was co-author of the technical report entitled "TECHNICAL REPORT AND MINERAL RESOURCE ESTIMATE FOR THE FENELON MINE PROJECT (according to National Instrument 43-101 and Form 43-010F)", effective date of July 5, 2016, and signature date of July 29, 2016 (revised on August 17, 2016), prepared for Wallbridge Mining Company Ltd.I am not aware of any material fact or material change with respect to the subject matter if the Technical Report that is not reflected in the Report, the omission of which would make the Technical Report misleading.
9. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
10. I have read NI 43-101 Respecting Standards of Disclosure for Mineral projects and Form 43-101F1, and the Technical Report has been prepared in accordance with that instrument and form.

Dated this 3rd day of March 2017, in the city of Val-d'Or (Québec)

(Original signed and sealed)

Catherine Jalbert, P.Geo.
InnovExplo Inc.
catherine.jalbert@innovexplo.com

CERTIFICATE OF AUTHOR – DENIS GOURDE

I, Denis Gourde, Eng (OIQ no.43860;) do hereby certify that:

1. I am a Consulting Engineer of: InnovExplo, 560, 3e Avenue, Val-d'Or, Québec, Canada, J9P 1S4.
2. I graduated with a B. Sc. degree in 1987 from the École Polytechnique of Montreal in 1987.
3. I am a member of the Ordre des Ingénieurs du Québec (OIQ, no. 43860).
4. I have worked as a mining engineer for a total of 27 years since obtaining my B.Sc. degree. My relevant experience for the purpose of the Technical Report is mainly:
    - VP Engineering & Sustaining Development, InnovExplo (2013–Present)
    - Corporate Director Community Affairs, Agnico Eagle (2011–2013)
    - General Manager, Agnico Eagle Meadowbank Mine (2007–2011)
    - General Manager, Cambior IAMGOLD (1998–2007)
5. I have read the definition of "qualified person" set out in Regulation 43-101 / 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.
6. I am author of and responsible for section 19 of the technical report entitled "NI 43-101 TECHNICAL REPORT ON THE PRE-FEASIBILITY STUDY FOR THE FENELON MINE PROPERTY (according to National Instrument 43-101 and Form 43-010F)", effective date of February 2, 2017, and signature date of March 3, 2017, prepared for Wallbridge Mining Company Ltd.
7. I had no prior involvement with the property that is the subject of the Technical Report.
8. I have not conducted a site visit of the property that is the subject of the Technical Report.
9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
10. I am independent of the issuer applying all of the tests in Section 1.5 of Regulation 43-101 (National Instrument 43-101).
11. I have read Regulation 43-101 respecting standards of disclosure for mineral projects and Form 43-101F1, and the sections of the Technical Report for which I was responsible have been prepared in accordance with that regulation and form.

Dated this 3rd day of March 2017, in the city of Val-d'Or (Québec)

(Original signed and sealed)

Denis Gourde, Eng.
InnovExplo Inc.
denis.gourde@innovexplo.com

CERTIFICATE OF AUTHOR – PIERRE PELLETIER

I, Pierre Pelletier, a registered mining engineer in the Province of Quebec (OIQ licence No. 36825), do hereby certify that:

1. I reside at 642 Mitchell Avenue, Mont Royal, Québec, Canada, H3R 1L4;
2. I am employed as a metallurgist by and carried out this assignment for InnovExplo Inc. – Consulting Firm in Mines and Exploration, 560, 3e Avenue, Val-d'Or, Québec, Canada, J9P 1S4
3. I graduated with a Bachelor of mining degree (B.Eng. Mining) from Université Laval in the city of Québec in 1982.
4. I have practiced my profession of mining engineer in mineral processing continuously for the last thirty-four (34) years in the fields of gold, base mineral and oxide mineral processing;
5. I am a member of the Ordre des Ingenieurs du Québec (OIQ licence No. 36825) and I am a member of the Canadian Institute of Mining and Metallurgy Association.
6. I have worked at different mineral processing positions, from process metallurgist, mill manager to vice-president metallurgy, for a total of 34 years since graduating from university. I acquired my mineral processing expertise with Falconbridge Copper (1983 to 1988), Audrey Resources (1988 to 1992), Barrick Gold (1992 to 2002), Cambior Inc. (2003 to 2006), and IAMGOLD corp. (2007-2015). I have been an independent consultant since 2015, working on specific mandates for InnovExplo since 2016.
7. I have read the definition of "qualified person" set out in Regulation 43-101 / 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.
8. I am author and responsible for sections 13, 17, and I am co-author of and also share responsibility for sections 1, 25, 26 and 27of the technical report entitled "NI 43-101 TECHNICAL REPORT ON THE PRE-FEASIBILITY STUDY FOR THE FENELON MINE PROPERTY", prepared for Wallbridge Mining Company Ltd. The effective date of the report is February 2, 2017, and the signature date is March 3, 2017.
9. I have not had any prior involvement with the project that is the subject of the Technical Report.
10. I have not conducted a site visit of the property.
11. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission of which would make the Technical Report misleading.
12. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
13. I have read NI 43-101 Respecting Standards of Disclosure for Mineral projects and Form 43-101F1, and the Technical Report has been prepared in accordance with that instrument and form.

Dated this 3rd day of March 2017, in the city of Longueuil (Québec)

(Original signed and sealed)

Pierre Pelletier, P.Eng.
InnovExplo Inc.
Pierrepelletier555@gmail.com

CERTIFICATE OF AUTHOR – MARIE-CLAUDE DION ST-PIERRE

This certificate applies to the NI 43-101 Technical Report for the Fenelon Mine Property prepared for Wallbridge Mining Company Ltd. issued on March 3, 2017 (the "Technical Report") and effective February 2, 2017.

I, Marie-Claude Dion St-Pierre, Eng., M.A.Sc. (OIQ licence No. 140947), do hereby certify that:

1. I am a Project Director with WSP Canada Inc. with a business address at 5355 boul. des Gradins, Québec (Québec) Canada, G2J 1C8.
2. I graduated from Sherbrooke University with a Bachelor's degree in Chemical Engineering in 2004 and Master's degree in Chemical Engineering in 2007.
3. I am a member of the Ordre des Ingénieurs du Québec (OIQ #140947).
4. I have worked as an engineer for a total of ten (10) years since graduating from my Bachelor's degree. My mining expertise was acquired while working for GENIVAR and Les mines Agnico-Eagle Ltée. I have been a consulting engineer for WSP Canada Inc. since January 2014.
5. I have read the definition of "qualified person" set out in National Instrument 43-101/Regulation 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;
6. I am responsible for the preparation of section 20 and I am co-author of sections 1, 25, 26 and 27 of the technical report entitled "NI 43-101 TECHNICAL REPORT ON THE PRE-FEASIBILITY STUDY FOR THE FENELON MINE PROPERTY", prepared for Wallbridge Mining Company Ltd. The effective date of the report is February 2, 2017, and the signature date is March 3, 2017.
7. I have not visited the Fenelon Mine Property for the purpose of the Technical Report.
8. I have not had any prior involvement with the project that is the subject of the Technical Report.
9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
10. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.
11. I have read NI 43-101 respecting standards of disclosure for mineral projects and Form 43-101F1, and the sections of the Technical Report for which I was responsible have been prepared in accordance with that regulation and form.

Dated this 3rd day of March 2017, in the city of Québec (Québec)

(Original signed and sealed)

Marie-Claude Dion St-Pierre, Eng. M.A.Sc.
WSP Canada Inc.
marie.claude.dion@wspgroup.com

CERTIFICATE OF AUTHOR – GEORGE DARLING

I, George Darling, Eng (PEO No. 10497014) do hereby certify that:

1. I am a Consulting Engineer for SNC-Lavalin 40 Larch St, Sudbury, Ontario, Canada.
2. I graduated with a Bachelor's degree in Mining Engineering from Queen's University in Kingston in 1976.
3. I am a member of the Professional Engineers of Ontario (PEO No. 10497014), and the Canadian Institute of Mines (149476).
4. I have worked as an engineer for a total of forty (40) years since graduating from university. My mining expertise was acquired while working for Inco INC and SRK, Stantec and SNC-Lavalin.
5. I have read the definition of "qualified person" ("QP") set out in National Instrument 43-101/Regulation 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 QP for the purposes of NI 43-101.
6. I am the author of items 15, 16, 18, 21 and 22 and co-author of items 1, and 25 to 27, of the report titled "NI 43-101 Technical Report on the Pre-feasibility Study for the Fenelon Mine Property" (the "Technical Report" or the "report"), prepared for Wallbridge Mining Company Ltd. The effective date of the report is February 2, 2017, and the signature date is March 3, 2017.
7. I have had prior involvement with the property that is the subject of the Technical Report.
8. I visited the Fenelon Mine Property for the purpose of the Technical Report on October 12, 2016.
9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
10. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
11. I have read NI 43-101respecting standards of disclosure for mineral projects and Form 43-101F1, and the items of the Technical Report for which I was responsible have been prepared in accordance with that instrument and form.

Signed on this 3rd day of March, 2017 in the city of Sudbury (Ontario)

(Original signed and sealed)

George Darling, P.Eng.
SNC-Lavalin
George.Darling@snclavalin.com

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1 SUMMARY

1.1 Introduction

At the request of Marz Kord, President and CEO of Wallbridge Mining Company Ltd ("Wallbridge", the "issuer" or the "Company"), InnovExplo Inc. ("InnovExplo") was retained to prepare a Technical Report (the "Report") to present and support the results of a Pre-feasibility Study ("PFS") for the Fenelon Gold Mine Property in accordance with Canadian Securities Administrators' National Instrument 43-101. InnovExplo is an independent mining and exploration consulting firm based in Val-d'Or, Québec.

The Fenelon Mine Property is an advanced stage project with near-term production potential, and drill intersections that suggest an exploration potential for resource expansion. The Property is situated near the Sunday Lake Deformation Zone, which hosts the Detour Lake mine in Ontario, belonging to Detour Gold Corporation, as well as the Martiniere gold project in Québec, held by Balmoral.

Wallbridge announced the results in a press release on February 2, 2017. The 2017 PFS estimates a Pre-tax Net Cash Flow of $6.62M and a Project pre-tax Internal Rate of Return ("IRR") of 92% for the initial approximate 18-month mine life for the known reserves located above 100 metres depth and in close proximity to the existing ramp.

The qualified persons responsible for the preparation of the Report included Pierre-Luc Richard, P.Geo. (InnovExplo), Bruno Turcotte, P.Geo. (InnovExplo). Catherine Jalbert, P.Geo (InnovExplo), Denis Gourde, P.Eng (InnovExplo), Pierre Pelletier, P.Eng. (InnovExplo), George Darling, P.Eng (SNC-Lavalin inc.) and Marie-Claude Dion St-Pierre, P.Eng (WSP Canada inc.).

The authors believe the information used to prepare the Report and to formulate its conclusions and recommendations is valid and appropriate considering the status of the Project and the purpose for which the Report is prepared. The authors, by virtue of their technical review of the Project's production potential, affirm that the work program and recommendations presented herein comply with National Instrument 43-101 and CIM technical standards.

1.2 Property Description and Location

The Fenelon Property is located in the Nord-du-Québec administrative region in the province of Québec (Canada), approximately 75 kilometres west-northwest of the city of Matagami.

The Fenelon Mine Property currently consists of one block of nineteen (19) mining claims staked by electronic map designation ("map-designated cells") and one (1) mining lease, for an aggregate area of 1,051.77 ha (10.5 km^2; Fig. 4.2). All mining titles are registered 100% in the name of Wallbridge Mining Company Ltd. All mining titles are in good standing according to the GESTIM database.

A net smelter royalty (NSR) of 1% is payable from production on the Fenelon Mine Property to Cyprus Canada Ltd, and and NSR Royalty of 1% is payable from production on the Fenelon Mine Property to Balmoral Ressources Ltd.

1.3 Geology

The Fenelon Mine Property is located in the northwestern Archean Abitibi Subprovince in the southern Superior Province of the Canadian Shield. The Abitibi Greenstone Belt is mainly composed of volcanic units unconformably overlain by large sedimentary Timiskaming-style assemblages. The metamorphic grade in the Abitibi Greenstone Belt generally varies from greenschist to subgreenschist facies, except in the vicinity of most plutons where the metamorphic grade corresponds mainly to amphibolite facies.

The Fenelon Mine Property lies within the Harricana-Turgeon volcano-sedimentary segment. The segment extends from the Detour Lake mine (Ontario) in the west to Matagami (Québec) in the east, and includes the Matagami, Brouillan, Joutel and Casa-Berardi mining districts. The two major northernmost faults of the Abitibi are the Sunday Lake (SLDZ) and Grasset (GDZ) deformation zones. The GDZ is the equivalent of the South Detour Deformation Zone in Ontario. The SLDZ and the GDZ are the major structural features in the area. They can be traced over 150 kilometres from the western boundary of the Abitibi Subprovince in Ontario, to the east of the Fenelon Mine Property and to the north of the Matagami mining camp. These two faults share many characteristics with others major breaks of the Abitibi.

The Fenelon Mine Property is covered by 4 to 50 metres of glacial overburden consisting mainly of sandy and gravel outwash material and lesser boulder-rich tills. There are no natural rock outcrops in the area of the Discovery Zone where glacial overburden is generally 4 to 8 metres thick. Detailed property-scale geological information is available for this area only where the bedrock has been drilled or exposed during open pit sampling and underground development work. The correlation between geological information and geophysical maps has contributed to the recognition of certain units based on magnetic signatures, such as magnetic-high gabbroic and ultramafic rocks, magnetic-low magnetic sedimentary rocks and highly conductive graphitic horizons.

The Discovery Zone is hosted in a series of siliceous zones and small-scale silica-albite shear zones within coarse-grained mafic intrusives that are segmented by a series of mafic dykes, between two panels of argillaceous sediments.

1.4 Mineralization

Gold mineralization is associated with a corridor of intense alteration located close to the contact between sediments and coarse-grained mafic intrusives, and within a coarse-grained mafic intrusive. The general orientation and dip of the silicified and mineralized envelopes is subparallel to the contact of the sediments and the coarse-grained mafic intrusives. Local variations in orientation and dip are present. The thickness of these envelopes varies from a few centimetres to 15 metres.

Gold mineralization is concentrated in the silicified envelopes and is associated with sulphides such as pyrrhotite, chalcopyrite and pyrite. Native visible gold is fairly common in drill hole intersections and in the wall rocks of developments. The grain size of the visible gold can reach 4 millimetres.

The mineralization described above occurs in two distinct styles and two distinct stages in the Discovery Zone, predominantly within a wide corridor delimited by the extent of the coarse-grained mafic intrusives. The mineralization styles are as follows:

Style 1: Early massive, laminated or brecciated silica-sulphide zones occurring along mafic dyke contacts, or commonly as isolated, irregular, metre-scale lensoid bodies inside the mafic dyke complex, like xenoliths of mineralized zone in the coarse-grained mafic intrusion. Pyrrhotite and pyrite are the dominant sulphides and occur as narrow fracture fillings or disseminations in silica-rich rock.

Style 2: Late narrow, lenticular or commonly tabular zones of silica-sulphide sericite alteration associated with small-scale (1–30 cm) shear zones occurring primarily along narrow dyke contacts. Sulphides occur disseminated in the altered rock or in quartz veinlets. The dominant sulphides are pyrrhotite, pyrite and chalcopyrite, with local coarse visible gold.

1.5 Status of Exploration and Drilling

In all, more than 50,000 metres have been drilled on the Fenelon Mine Property, and two bulk samples have been mined and processed from the deposit. In 2001, a 13,713-tonne bulk sample mined from a small open pit was test-milled at the Camflo Mill in Malartic. The sample returned 132,039 grams (4,245 oz) of gold for a reconciled head grade of 9.84 g/t Au using a calculated recovery of 97%. A second bulk sample, consisting of 8,073 tonnes mined from underground, was also milled at Camflo and returned 80,731 grams (2,596 oz) of gold for a reconciled head grade of 10.7 g/t Au and a gold recovery of 93.5%. Compensating for the operational problem that occurred during the ore processing of this second bulk sample, the gold recovery would have been in the range of 97%.

Prior to the 2016 mineral resource estimate, resources had last been estimated in September 2004 and updated in January 2005. About 16,000 metres of additional diamond drilling have been completed since that time. In 2016, Wallbridge initiated an exploration program on the Fenelon Mine Property. The first phase of the program involved a review of historical drilling in close proximity to the mine workings and additional sampling of previously unsampled historical drill core, where warranted. The results from the first three batches of samples included a sample with visible gold that assayed 89.3 g/t Au over 0.35 metre.

The issuer did not carry out any drilling on the Fenelon Mine Property.

1.6 Status of Development and Operations

The historical underground workings are currently flooded and the issuer did not conduct any underground operations on the Fenelon Mine property.

1.7 Mineral Processing and Metallurgical Testing

The Fenelon ore responds well to conventional gold leaching with gold recoveries of up to 98-99 % in the limited laboratory testwork done to date. The two previously tested bulk samples returned gold recoveries of 97% and 93.5%, although operational problems were reported on both occasions.

The additional testwork done in 2016 on historical core samples from the Fenelon deposit failed to confirm the gold recoveries by direct cyanidation. However, intensive leaching on the leach tails from those tests returned similar high gold recoveries of up to the 98-99% at the target grind size. Considering the results to date on the bulk and laboratory samples, the gold recovery of 97 % appears appropriate at this stage. Test work will continue when new samples become available.

The results from the testwork done to date in laboratory and during the processing of two bulk samples in commercial plant propose that the preferential way to process the Fenelon ore should be the conventional gold leaching process. In the current situation in the Abitibi area, there exist some possibilities for competitive quotations from different processing facilities. For the purpose of this study, it is considered that the ore will be processed at the same facility as the previous two bulk samples. The process facility used was the Camflo Mill located in the Malartic town area and using a Merrill Crowe process.

1.8 Mineral Resource Estimate

In 2016, InnovExplo was mandated to prepare a mineral resource estimate on the Fenelon deposit and a supporting Technical Report in accordance with National Instrument 43-101 (“NI 43-101”) and Form 43-101F1. A model was generated for the entire drilled area of the Fenelon deposit based on all available geological information and analytical results.

The 2016 Fenelon Deposit Mineral Resource Estimate (the “2016 MRE”) was prepared by Pierre-Luc Richard, P.Geo., and Catherine Jalbert, P.Geo., using all available information.

The 2016 resource area measures 500 metres along strike, 210 metres wide and 280 metres deep. The resource estimate was based on a compilation of historical and recent diamond drill holes and wireframed mineralized zones, largely inspired by previous work and Wallbridge’s interpretation.

Given the density of the processed data, the search ellipse criteria, the drill hole density and the specific interpolation parameters, InnovExplo classified the 2016 MRE as Measured, Indicated and Inferred resources. The estimate is compliant with CIM standards and guidelines for reporting mineral resources and reserves.

Following a detailed review of all pertinent information and after completing the 2016 MRE, InnovExplo concluded the following:

- Geological and grade continuity have been demonstrated for eight (8) of the nine (9) mineralized zones composing the Fenelon deposit. The ninth zone was not attributed to any resource.
- Using a cut-off grade of 5.00 g/t Au, the Measured Resources stand at 30,100 tonnes grading 13.12 g/t Au for 12,700 ounces of gold, the Indicated Resources stand at 61,000 tonnes grading 12.89 g/t Au for 25,300 ounces of gold, and Inferred Resources stand at 6,500 tonnes grading 9.15 g/t Au for 1,900 ounces of gold.
- It is likely that additional diamond drilling would upgrade some of the Inferred Resources to Indicated Resources.

- It is likely that additional diamond drilling would identify additional resources down-plunge and in the vicinity of known mineralization.

The following table presents the results of the In Situ Mineral Resource Estimate for the Fenelon deposit.

Fenelon Deposit Mineral Resource Estimate at a 5.00 g/t Au cut-off grade (Table 14.8)

> 5.00 g/t Au		Tonnes (t)	Au (g/t)	Contained Au (oz)
Measured (M) and Indicated (I)	Measured (In-situ)	27,000	13.94	12,100
	Measured (broken)	3,100	6.14	600
	Indicated	61,000	12.89	25,300
	Total M+I	91,100	12.97	38,000
Inferred	In-situ	6,500	9.15	1,900

- The Independent and Qualified Persons for the Mineral Resource Estimate, as defined by NI 43-101, are Pierre-Luc Richard, P.Geo., M.Sc. and Catherine Jalbert, P.Geo., B.Sc., of InnovExplo Inc. The effective date of the estimate is July 5, 2016.
- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
- The model includes nine gold-bearing zones, eight of which include resources at the official cut-off grade.
- Results are presented in situ and undiluted.
- Sensitivity was assessed using cut-off grades from 2.00 to 10.00 g/t Au, with 1.00 g/t Au increments. The official resource is reported at a cut-off of 5.00 g/t Au. Cut-off grades must be re-evaluated in light of prevailing market conditions (gold price, exchange rate and mining cost).
- A fixed density of 2.80 g/cm^3 was used for all zones, supported by limited information.
- A minimum true thickness of 2.0 metres was applied, using the grade of the adjacent material when assayed, or a value of zero when not assayed.
- High grade capping (Au) was applied to raw assay data and varies from 30 g/t to 140 g/t based on the statistical analysis of individual mineralized zones. Restricted search ellipsoids were used during interpolation using 1X variography ranges and a threshold of 30 g/t Au.
- Compositing was done on drill hole intercepts falling within the mineralized zones (composite lengths vary from 1 to 3 metres in order to distribute the tails adequately).
- Resources were evaluated from drill holes using a 2-pass ID3 interpolation method in a block model (block size = 5 m x 5 m x 5 m).
- The inferred category is only defined within the areas where blocks were interpolated during pass 1 or pass 2 where continuity is sufficient to avoid isolated blocks being interpolated by only one drill hole. The indicated category is only defined by blocks interpolated by a minimum of two drill holes in areas where the maximum distance to the closest drill hole composite is less than 20 metres for blocks interpolated in pass 1. The measured category is only defined by blocks interpolated by a minimum of two drill holes in areas where the maximum distance to the closest drill hole composite is less than 20 metres for blocks interpolated in pass 1 and in close proximity with sampled drifts (<10 metres).
- Ounce (troy) = metric tons x grade / 31.10348. Calculations used metric units (metres, tonnes and g/t).
- The number of metric tons was rounded to the nearest hundred. Any discrepancies in the totals are due to rounding effects. Rounding followed the recommendations in NI 43-101.
- InnovExplo is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, marketing or other relevant issue that could materially affect the Mineral Resource Estimate.

1.9 Mining plan and Mineral Reserves

The underground mine design for the Fenelon deposit will provide for a 13-month mine life, from initial underground mine rehabilitation to completion of mining. The mining plan was developed using the Indicated and Measured Mineral Resources estimate provided by InnovExplo. The Proven and Probable Mineral Reserves within the underground mine include 96,721 tonnes of ore at an average grade of 9.3 g/t Au after dilution and mining recovery factors are applied.

Prior to the commencement of mining activities, the existing open pit and underground workings should be dewatered according to local regulations and the workings would need to be rehabilitated to allow the new development to start. The mining activities would be spread over a total of six (6) levels, from which two (2) are currently partially developed and would require rehabilitation. The remaining 4 levels would need to be developed.

Based on the nature and geometry of the Fenelon deposit, three different mining methods should be employed: long hole, uppers, and drift and fill.

Tables below present a summary of the Fenelon deposit mineral reserves and the mine plan metrics.

Mineral Reserves Summary – Fenelon Deposit (Table 25.2)

Category	Mined Tonnes	Diluted and Recovered Tonnes	Grams Recovered	Ounces
Proven	6,321	6,770	62,970	2,025
Probable	83,974	89,951	836,600	26,897
Total	**90,295**	**96,721**	**899,570**	**28,922**

Mining metrics (Table 25.3)

Mining metrics							Total
Type		Q4 2017	Q1 2018	Q2 2018	Q3 2018	Q4 2018	
Ore production (DMT)							
	Ore	1,789	30,997	29,513	29,208	5,216	96,721
	Total	**1,789**	**30,997**	**29,513**	**29,208**	**5,216**	**96,721**
Horizontal development (m)							
	Drift - Level Access	130	227	-	-		357
	Drift and Fill Dev	-	33	58	35		127
	Sill - Rock	184	294	246	-		724
	Sill - Ore	51	220	157	-		428
	Ramp	232	260	-	-		491
	Total	**597**	**1,035**	**460**	**35**		**2,127**
Vertical development (m)							
	Raise Development	-	50	-	-		50
	Total	**-**	**50**	**-**	**-**		**50**

1.10 Project Infrastructure

Due to the nature of the project, minimal infrastucture would be required on site during the project execution. The surface infrastructure would be limited to the upgrade of existing access roads, enlargement of the existing settling pond, construction of a camp equipped with dry facilities, offices, an ore crusher and surface mining infrastructure such as ventilation and heaters, compressor and power generation. A surface garage and shops would be established inside the only structure currently existing on site. Underground infrastructure such as dewatering, refuge station, escapeway, power distribution, explosive magazine and ventilation will also be constructed by a mining contractor to support the development and operations activities. No ore processing facility would be built on site.

1.11 Environmental Studies, Permitting and Social or Community Impact

The available information taken from previous authorization requests and public data base and documents, was used to establish the current environemental state of the Fenelon Mine Property. The on-site investigations that were done were on the surface water quality, sediment quality and on the fish community.

A limited amount of waste rock, approximately 60,000 tonnes, will be generated during the mine development. Due to the production and the mining sequence, a surface waste rock pile is required.

Underground water management is defined by the two main project phases: the initial dewatering of the open pit and existing underground excavation, and the ongoing dewatering during the development and operation phases. The water will be treated through a polishing pond and discharged to the environement.

Run-off water from the waste rock, overburden and ore piles will either be collected by a system of ditches and conveyed to the open pit or gravitationally directed towards the pit.

Fenelon Mine Property falls under the Northern Québec regime for the environmental and social impact assessment and review procedure. Wallbridge is required to submit an environmental impact assessment to the. No formal EIA is currently needed under federal regulations.

Site restoration costs are estimated at CAD $ 989 869, which include the direct and indirect costs, as well as post-closure monitoring. Wallbridge will have to provide a financial guarantee whose amount corresponds to 100% anticipated cost. A closure and rehabilitation plan will have to be submitted and approved by the MERN prior to production activites.

The Fenelon Mine Property is located in the territory of the Eeyou Istchee James Bay Regional Government. The sector appears to be used very little by the neighboring communities because the ecological characteristics of the territory limit the potential for use and development. The proponent's intended meetings with the relevant Aboriginal communities and local stakeholders and ensure regular communication with the communities. The consultation activities will be aimed at informing and consulting people living in the territory throughout the process, from project planning to the end of the exploitation of the mine. At meetings held in the past, there was much talk about employability, not only in terms of onsite jobs and contractors, but also the hiring of specialized firms with which the Crees have partnerships. The environment was also mentioned as a priority issue.

1.12 Capital and Operating Costs

The construction and operational strategy for the mining project on the Fenelon Mine Property relies on the use of contractors. As opposed to larger projects, which are often broken into sharply defined construction and operations phases, the proposed strategy for Fenelon is to hire contractors at the start of the Project and continue the collaboration until the end of the mine life. This is made possible due to the nature, duration and scale of the Project. The project cost estimate was developed considering that contracts would be required for on-site activities, such as initial site preparation and settling pond construction, dewatering, mine development and production, surface and underground construction, ore crushing, surface buildings and camp construction as well as for operation.

It is not planned to build ore processing facilities on site; instead, the ore would be trucked to an existing facility outside the property. This strategy was used when developing the cost estimate.

The estimated pre-tax capital and operating expenditures are summarized in the following table, distributed by quarter and grouped under the main categories.

Cost expenditure summary for the mining project on the Fenelon Mine Property (C$ '000) (Figure 25.4)

Cost Item	Q1 2017	Q2 2017	Q3 2017	Q4 2017	Q1 2018	Q2 2018	Q3 2018	Q4 2018	Total
Pre-production	300	668	80	0	0	0	0	448	1,496
Capital costs	0	103	1,588	1,766	1,728	53	0	0	5,238
Operating costs	0	105	1,120	3,124	7,838	5,845	4,982	697	23,710
Remote camp operations	0	0	678	400	537	537	450	438	3,041
General and Administrative	0	0	299	433	567	702	567	299	2,866
Contingency	29	66	376	560	1,049	705	593	239	3,616
Royalties	0	0	0	8	248	289	234	31	809
Total	329	941	4,140	6,292	11,968	8,131	6,825	2,152	40,777

The estimate includes contingency, which represents 9.9% of the total cost before contingency. This percentage was determined by evaluating the quantity and cost precision of each system element of the cost estimate. As a result, contingency by item varies between 5.6% and 50%. The cost estimate accuracy falls within -3% to +18%.

1.13 Financial Analysis

Based on the current assumptions, discounted cash flow modelling of the project yields a pre-tax NPV of C$5.84 million at a 5% discount and a pre-tax internal rate of return ("IRR") estimate of 92%. The NPV and IRR, after income taxes and before any withholding tax, are C$2.80 million and 60%, respectively. A summary of the results is presented in the table below.

Base case estimated financial results (Table 25.5)

PRE-TAX	
NPV at 5% Discount Rate (C$ '000)	5,842
Internal Rate of Return (IRR)	92%
Payback Period	Q3 2018
AFTER-TAX	
LOM NPV at 5% Discount Rate (C$ '000)	2,802
Internal Rate of Return (IRR)	60%
Payback Period	Q2 2018

1.14 Interpretation and Conclusions

The objective of the author's mandate was to prepare a Technical Report (the "Report") to present and support the results of a Pre-feasibility Study ("PFS") for the Fenelon Gold Mine in accordance with Canadian Securities Administrators' National Instrument 43-101 Respecting Standards of Disclosure for Mineral Projects ("NI 43-101") and Form 43-101F1.

The authors consider the present Pre-feasibility Study (and Resource Estimate herein) to be reliable and thorough, based on quality data, reasonable hypotheses, and parameters compliant with NI 43-101 and CIM standards regarding mineral resource estimates.

The issuer's Fenelon Mine Property covers 1,052 hectares and is located in west-central Québec about 75 kilometres northwest of the town of Matagami. Geologically, it is situated near the Sunday Lake Deformation Zone, which hosts the Detour Lake mine in Ontario (Detour Gold Corporation) and the Martiniere gold project in Québec (Balmoral Resources Ltd). The Fenelon deposit (a.k.a. the Discovery Zone) has seen both underground and open pit development in the past.

1.14.1 Mineral Resources

Following a detailed review of all pertinent information and after completing the 2016 MRE, InnovExplo concluded the following:

- Geological and grade continuity have been demonstrated for eight (8) of the nine (9) mineralized zones composing the Fenelon deposit. The ninth zone was not attributed to any resource.
- Using a cut-off grade of 5.00 g/t Au, the Measured Resources stand at 30,100 tonnes grading 13.12 g/t Au for 12,700 ounces of gold, the Indicated Resources stand at 61,000 tonnes grading 12.89 g/t Au for 25,300 ounces of gold, and Inferred Resources stand at 6,500 tonnes grading 9.15 g/t Au for 1,900 ounces of gold.
- It is likely that additional diamond drilling would upgrade some of the Inferred Resources to Indicated Resources.
- It is likely that additional diamond drilling would identify additional resources down-plunge and in the vicinity of known mineralization.

1.14.2 Exploration Potential – 2016 Technical Report

Following a detailed review of all pertinent information and after completing the 2016 MRE, InnovExplo concluded the following in the 2016 Technical Report (Richard et al., 2016):

- Geological and grade continuity have been demonstrated for eight (8) gold-bearing zones on the Fenelon Mine Property;
- A large proportion of the resource is located in close proximity to existing underground workings at shallow depth;
- The bulk of the resource is located in the first 150 metres from surface (87% of the tonnes and 91% of the ounces);
- It is likely that additional diamond drilling would upgrade some of the Inferred Resources to Indicated Resources;
- There is the potential for upgrading some of the Indicated Resources to Measured Resources through detailed geological mapping, infill drilling and systematic channel sampling from the underground workings;
- A zone that was intercepted by four mineralized intervals (Zone 110) was modelled but not interpolated, and is considered as an exploration target which requires tighter drill spacing before it can be interpolated;
- There are several opportunities to add additional resources by drilling the depth extensions of the ore shoot that originates in the resource area and the subparallel mineralized zones in the vicinity of the currently identified zones; and
- A property-scale compilation and target generation program should be completed. Conversion drilling should be devoted to upgrading part of the Inferred resources to the Indicated category, whereas the objective of exploration drilling should be to target the currently identified ore shoots at depth and discover additional zones over the entire project.

1.14.3 2017 Pre-feasibility Study

The following sections present the interpretation and conclusions for each component of this Pre-Faisability Study.

1.14.3.1 Mining Plan and Mineral Reserves

- The mineral reserve estimate for the Fenelon deposit is based on the resource block model provided to Wallbridge by InnovExplo, along with information in the InnovExplo report titled “Technical Report and Mineral Resource Estimate for the Fenelon Mine Property”, dated August 17, 2016 (Richard and al. 2016).
- The underground mine design for the Fenelon deposit will provide for a 13-month mine life, from initial underground mine rehabilitation to completion of mining. The Proven and Probable Mineral Reserves within the underground mine include 96,721 tonnes of ore at an average grade of 9.3 g/t Au after dilution and mining recovery factors are applied.
- Prior to the commencement of mining activities, the existing open pit and underground workings should be dewatered according to local regulations and the workings would need to be rehabilitated to allow the new development to start. The mining activities would be spread over a total of six (6) levels, from which two (2) are currently partially developed and would require rehabilitation. The remaining 4 levels would need to be developed.
- Based on the nature and geometry of the Fenelon deposit, three different mining methods should be employed: long hole, uppers, and drift and fill.

1.14.3.2 Metallurgy

- Considering the results to date on the bulk and laboratory samples, a gold recovery of 97% appears appropriate at this stage for this study. However, it will be safe to proceed with confirmation testwork when new samples become available.
- In the future, it will be critical to control liquid losses during ore processing; otherwise, the final gold recovery will be negatively affected as it was during the processing of the 2004 bulk sample.
- It may be appropriate to track the copper grades and optimize ore mixing to control the copper grade and sulphide variations in the mill feed.
- The CIL gold recovery process may be a viable alternative to the current Merrill Crowe process.

1.14.3.3 Environment

- The available information for the Fenelon Mine Property does not reveal any critical element that could seriously affect the future development of the project. Additional studies will have to be conducted in order to complete an environmental baseline.

- A consultation plan will be developed to assess the perceptions of the Project by Cree, Algonquin and Jamesian communities, and to identify appropriate mitigation measures.

- To move forward with the Project, Wallbridge is required to submit an EIA for the Project to the Review Committee (COMEX). No formal EIA is currently needed under federal regulations.

- The EIA process is currently underway and began with the submission of the Project's preliminary information to the Evaluating Committee (COMEV) in November 2016. A directive should be issued early in 2017.

- According to previous documents, the waste rock was considered not potentially acid generating with only a low leachability in Cd and Ba. However, when compared to current criteria, the waste rock is leachable in Ba, Cd, Cu, Mn, Ni and Zn.

- As for the ore, previous documents indicate the results of all samples submitted to the static acid generation potential tests fall in the uncertainty zone. The ore is also leachable in Cd, Cu and Mn.

- According to geological data, ore rocks are associated with silicification and the most abundant sulphides would be pyrrhotite (trace to 30%) and pyrite. Since pyrrhotite is the most reactive sulphide capable of causing acid mine drainage (AMD), it is recommended that the geochemical characterization be enhanced.

- A conceptual closure plan will have to be prepared with respect to the "*Guide de préparation de réaménagement et de restauration des sites miniers au Québec*" published in 2016. It will outline measures to be taken for progressive rehabilitation, closure rehabilitation and post-closure monitoring. It will also help refine the evaluation of restoration costs completed as part of this Report.

- The conceptual plan has to be presented to the MERN for approval before the beginning of the mining activities.

1.14.3.4 Capital and operating costs

- The construction and operational strategy for the mining project on the Fenelon Mine Property relies on the use of contractors.

- The project cost estimate was developed considering that contracts would be required for on-site activities, such as initial site preparation and settling pond construction, dewatering, mine development and production, surface and underground construction, ore crushing, surface buildings and camp construction as well as for operation.

- It is not planned to build ore processing facilities on site; instead, the ore would be trucked to an existing facility outside the property. This strategy was used when developing the cost estimate.

- The estimated pre-tax capital and operating expenditures are estimated at C$40,777 M.

- The estimate includes contingency, which represents 9.9% of the total cost before contingency. This percentage was determined by evaluating the quantity and cost precision of each system element of the cost estimate. As a result, contingency by item varies between 5.6% and 50%. The cost estimate accuracy falls within -3% to +18%.

1.14.3.5 Financial analysis

Based on the current assumptions, discounted cash flow modelling of the project yields a pre-tax NPV of C$5.84 million at a 5% discount and a pre-tax internal rate of return ("IRR") estimate of 92%. The NPV and IRR, after income taxes and before any withholding tax, are C$2.80 million and 60%, respectively. A summary of the results is presented in the table below.

Base case estimated financial results (Table 25.5)

Pre-tax	
NPV at 5% Discount Rate (C$ '000)	5,842
Internal Rate of Return (IRR)	92%
Payback Period	Q3 2018
After-tax	
LOM NPV at 5% Discount Rate (C$ '000)	2,802
Internal Rate of Return (IRR)	60%
Payback Period	Q2 2018

1.14.4 Conculsions

InnovExplo, SNC-Lavalin and WSP conclude that the 2017 Pre-feasibility Study presented herein allows the project on the Fenelon Mine Property to advance to the production stage for which potential viability has been demonstrated.

1.15 Risks and Opportunities

The table below (Tab. 25.6) identifies the significant internal risks, potential impacts and possible risk mitigation measures that could affect the future economic outcome of the project on the Fenelon Mine Property. The list does not include the external risks that apply to all mining projects (e.g., changes in metal prices, exchange rates, availability of investment capital, change in government regulations, etc.). Significant opportunities that could improve the economics, timing and permitting are identified in the following table (Tab. 25.7). Further information and study is required before these opportunities can be included in the project economics.

Risks for the Fenelon mining project (Table 25.6)

Expertise	Risk	Potential Impact	Possible Risk Mitigation
Metallurgy	Metallurgical recoveries are based on limited testwork	Recovery might differ negatively from what is currently being assumed	Conduct additional metallurgical tests
Metallurgy	Operational problems occurred during the two bulk sample processing campaigns	Gold was affected negatively in the second bulk sample by 3.5% (97% to 93.5%.) The effect on the first sample was not clear.	Operational problems could occur again in the future. Attention will need to be taken regarding the gold recovery process to understand the source of the problem and find a solution. CIL may prove to be a more viable process than Merrill Crowe.
Metallurgy	Ore samples used in the last characterization test (2016) were old, providing mixed results	May not be representative in terms of quality. Gold kinetics were very slow. It was not determined whether this was due to the state of the sample or another property.	Additional testwork will need to be done when new samples or ore become available.
Mining	Conditions of the existing underground mine are unknown.	The rehabilitation process may be a financial risk as well as a scheduling risk	Monitoring during the early stages of rehabilitation.
Mining	Slope conditions of the open pit while it is being dewatered are not yet known.	Rehabilitation of the slopes may be required	Monitoring during the early stages of dewatering.
Mining	Mine dewatering requirements during the operations phase have not been fully quantified	Mine water volumes may exceed calculations, leading to greater than expected demands on water management and the dewatering system.	Upgrade the existing surface water pond. Use the existing pit bottom for water management. Monitoring during the early stages of the operation and modify the dewatering system if required.
Mining	Selbaie road maintenance costs	Several users currently share the costs of Selbaie road maintenance. There is a risk that some users would no longer use this road, resulting in higher maintenance costs for the road leading to the Fenelon mine	NA
Environment	Insufficient or incomplete environmental studies or baseline data	Field work to comply with new guidelines. Higher CAPEX cost. Delay of the EIA submission, and thus the mine schedule.	Studies and field work should be performed during early stage of the EIA process.

Expertise	Risk	Potential Impact	Possible Risk Mitigation
Environment	Project located within a priority sector for the creation of a protected area for woodland caribou.	Longer analysis by the ministry, and thus a delay in the mine schedule	Early discussion with the ministry on possible mitigation measures.
Environment	Waste rock acid-generating and metal leaching	Higher CAPEX and OPEX cost for management. Higher cost for post-rehabilitation monitoring	No waste rock piled on surface
Environment	First Nations and/or social issues	Delay of the Project's social acceptance, and thus a delay in the mine schedule.	Hold meetings with stakeholders early during project development to address major issues and elaborate mitigation measures

Opportunities for the Fenelon mining project (Table 25.7)

Expertise	Opportunities	Explanation	Potential benefit
Geology	**Exploration potential**	Potential for additional discoveries at depth and around the Fenelon deposit by drilling Additional resources may be present in the immediate vicinity of the mine workings as demonstrated by the recent re-sampling program Additional resources identified by the delineation drilling of exploration targets 109 and 110, and by following zone extensions at depth	Potential to increase resources and extend mine life
Metallurgy	**Metallurgy**	Recovery might be better than what is currently being assumed	Potential to increase resources and improve the viability of the project
Metallurgy	**CIL process**	CIL could be an alternative process to avoid the liquid losses occurring in the Merrill Crowe process	Potential elimination of the operational problem and stable/better gold recovery
Metallurgy	**Gravity gold recovery**	Coarse gold recovery by gravity could potentially be a good process for this ore	Minimize potential gold losses and trapping in mill
Mining	**Cost and schedule**	Early receipt of dewatering permit.	The mine could be dewatered sooner, therefore yielding a better understanding of site conditions.

Expertise	Opportunities	Explanation	Potential benefit
Mining	**Cost and schedule**	Dewatering of the underground mine can be done via the ramp or the existing raise	Dewatering process could be accelerated.
Environment	**EIA**	Use theoretical and/or existing data to complete environmental studies.	Lower CAPEX cost. Shorter delay in submitting the EIA, thus shorter delays in the mine schedule.
Environment	**Mine Closure**	Keep and re-use surface infrastructure for use during future exploration at the end of the LOM. Use the waste rock entirely as rock fill material in open stopes	Lower mine closure cost.
Financial	**Financial**	The Company is an exploration company and has the right to use loss carry forwards	The loss carry forwards can be applied to reduce income taxes. This has not been considered in the financial evaluation of the project

1.16 Recommendations

Based on the results of this Pre-feasibility Study, InnovExplo, SNC-Lavalin and WSP recommend advancing the project on the Fenelon Mine Property to the production stage.

1.16.1 Site exploration and development

In order to extend the mine life and the project's financial benefits, it is recommended that above and underground exploration drilling and development work be conducted from surface and from the existing ramp. Moreover, additional exploration work will allow underground conditions to be assessed, thereby refining the mine design and lowering the risks for mining operations. By doing so, dewatering could be executed earlier in the project schedule.

While exploration work is going on, complementary engineering and environmental studies could be completed simultaneously. This will help characterize the project and site conditions, and yield a more accurate impact assessment.

The environmental impact assessment and review procedure can be conducted during exploration work. However, the more confirmed details from complementary engineering and environmental studies and a more complete mine development, operation and closure description could facilitate and accelerate a ministry review and approval of the project.

The following sections detail the recommended two-phase work program:

- Phase 1 – Exploration work and complementary engineering and environmental studies; and
- Phase 2 – Mine development and operation.

1.16.2 Phase 1 – Exploration Work and complementary Engineering and Environmental studies

1.16.2.1 Exploration work

InnovExplo recommends that Wallbridge continue to revise the property-scale compilation and to generate targets. Additional drilling should target the down-plunge extensions of the currently identified mineralized zones as described in this Technical Report. An additional objective would be the discovery of additional zones of similar mineralization near the currently identified mineralized zones. If additional work proves to have a positive impact on the project, the current resource estimate should be updated.

In summary, InnovExplo recommends the following work program:

Phase 1a:

- Initiate a property-scale compilation and target generation program;
- Conduct infill and down-plunge exploration drilling aimed at expanding the current resources.

Phase 1b (after mine dewatering and contingent upon the success of Phase 1)**:**

- Follow-up underground drilling program on the Fenelon deposit to potentially add resources;
- Update the 3D model and resource estimate.

InnovExplo has prepared a cost estimate for the recommended work program to serve as a guideline for the Fenelon Mine Property. The budget for the proposed program is included in the table presented in section 1.16.3.3. Phase 1b is contingent upon the success of Phase 1a.

1.16.2.2 Environment

The following two sections present the recommended additional environmental work to obtain the required permits and to define the waste rock, ore and water management systems for the Fenelon Mine Project. These studies should be carried out between April and July 2017.

1.16.2.2.1 Baseline information

Additional environmental and social activities will be required to better assess the impacts of the project to be reported in the EIA. A preliminary list of these complementary activities is presented below. This list could be adjusted to meet the COMEV's directive requirements.

- Hydrological study, water surface quality, sediment quality;
- Hydrogeological study and underground water quality;
- Soil quality assessment;
- Air dispersion model;
- Waste rock and ore geochemical characterization; and
- Consultations with First Nations and stakeholders.

The environmental permitting costs, including the EIA, should be revised at the next stage of project development, as the extent of the requirements will be known following the COMEV's directive.

1.16.2.2.2 Ore, waste rock and water management

The next stage of geochemical characterization studies should be undertaken together with a refined assessment of waste rock quantity by lithology in order to support waste rock management options. Additional sampling and testing (static and kinetic tests) should be carried out.

A review of the surface water management infrastructure should be completed and the design should be updated during the next stage of engineering. A monitoring plan of the final effluent should be included in the surface water management plan.

The cost estimate for the environmental work is included in the table presented in section 1.16.3.3.

1.16.2.3 Metallurgy

Because additional drilling is recommended, metallurgical testwork is also recommended to confirm the gold recovery for the current and additional resources. The suggestion is to proceed with CIL testwork as an alternative to the Merrill Crowe process.

The cost estimate for the recommended work is included in the table presented in section 1.16.3.3.

1.16.2.4 Complementary Engineering Studies

Based on the results of this Pre-feasibility Study, it is recommended that the following work plan to be completed before commencing mining operations.

The following engineering studies should be completed before commencing mining operations:

- It is recommended that further geomechanical studies be done prior to commercial operation because the proposed bulk / longhole mining approach differs from Golder's more selective / cut & fill mining method. This study should include the final selection of the technology to be used for cemented fill.
- To design the polishing pond, it is recommended that more information be obtained on site topography, soil parameters, the existing polishing pond design (and performance) and water characteristics. This additional information can be used to update the design parameters presented in this document, if necessary. A site hydrology study is also recommended.
- At this point, no hydrogeological modelling was performed as part of the pre-feasibility study to quantify the dewatering requirements during operations. A hydrogeological model could be used to estimate the expected inflow during operations.

The estimated costs for the recommended work program are included in the table presented in section 1.16.3.3.

1.16.3 Phase 2 – Mine Development and Operation

1.16.3.1 Mining and infrastructure

At the start of the mine development, it is recommended that Wallbridge put together an owner team to work closely with and monitor the progress of the contractors working on site. The owner's team should ideally be multidisciplinary, but due to the scale of the project scale, it could be limited to essential positions. At a minimum, the team should include the following:

- Mine Manager;
- Mine Geologist;
- Mine Engineer;
- Process Engineer;
- Mine Safety and Training Officer;
- Site Security;
- Core Cutting and Sampling Technician.

Occasional consulting engineering work may be required by the owner's team during mine development and operations to support the contractor and owner teams. Costs for the owner's team and engineering support are included in the global project cost estimate.

1.16.3.2 Contractor mobilization

The site development strategy is based on the use of contractors. The owner should finalize contracts with the mining contractor and the camp contractor before the start of the exploration phase. The mining contractor would be in charge of installing the entire infrastructure required for dewatering, mine rehabilitation and drilling, as well as the drilling services. The camp contractor would be in charge of setting up the exploration camp.

The mill and ore transportation contracts should also be awarded to secure the milling capacity. Once the operations phase starts, the contractors would already be on site. This can be leveraged as a good opportunity for the project. The contracts should be awarded based on the owner's governance rules in order to be ready for exploration when the moving forward with the decision taken by Wallbridge.

1.16.3.3 Total Cost Estimate for Additional Work

The cost estimate for Phase 1 of the recommended work (additional exploration work and complementary engineering and environmental studies) is presented in the table below, for a total of C$3,780,000, including a 20% contingency.

SNC-Lavalin, WSP and InnovExplo are of the opinion that the recommended work program and proposed expenditures are appropriate and well thought out, and that the character of the Fenelon Mine Property is of sufficient merit to justify the recommended program. SNC-Lavalin, WSP and InnovExplo believe that the proposed budget reasonably reflects the type and amount of contemplated activities.

Estimated costs for the recommended work program (Table 26.1)

Phase 1 - Work Program: Exploration Work and complementary Engineering and Environmental studies	Budget	
	Description	Cost
Exploration		
Property-scale compilation and target generation		$25,000
Surface drilling on the Fenelon deposit (all-inclusive)	15,000 m	$1,500,000
Subtotal 1.		**$1,525,000**
Exploration		
Follow-up underground drilling on the Fenelon deposit (all inclusive)	10,000 m	$1,000,000
Update 3D model and resource estimate		$100,000
Subtotal 2.		**$1,100,000**
Environmental Studies		$200,000
Environmental permitting ($200,000, included in the cost expenditure summary of Table 21.2 at the production stage).		$200,000
Subtotal 3.		**$400,000**
Metallurgy		
Additional metallurgical test work on the current and additional resources and CIL testwork		$50,000
Subtotal 4.		**$50,000**
Complementary Engineering Studies		
Geomechanical & backfill study		$25,000
Polishing pond engineering ($50,000 included in the cost expenditure summary Table 21.2, at the production stage)		$50,000
Hydrology study (costs included in 2a) environmental studies)		$0
Hydrogeological study (costs included in 2a) environmental studies)		$0
Subtotal 5.		**$75,000**
Subtotal Phase 1		**$3,150,000**
Contingencies	20%	$630,000
Total Phase 1 Work Program		**$3,780,000**

2 INTRODUCTION

At the request of Marz Kord, President and CEO of Wallbridge Mining Company Ltd ("Wallbridge", the "issuer" or the "Company"), InnovExplo Inc. ("InnovExplo") was retained to prepare a Technical Report (the "Report") to present and support the results of a Pre-feasibility Study ("PFS") for the Fenelon Gold Mine Property in accordance with Canadian Securities Administrators' National Instrument 43-101 Respecting Standards of Disclosure for Mineral Projects ("NI 43-101") and Form 43-101F1.

InnovExplo is an independent mining and exploration consulting firm based in Val-d'Or (Québec). The report was prepared using contributions from SNC-Lavalin Inc. ("SNC-Lavalin"), an independent engineering and construction firm based in Canada with offices in Québec and other provinces, and WSP Canada Inc. ("WSP), an independent engineering professional services consulting firm also based in Canada with offices in Québec and other provinces.

2.1 Issuer

The issuer was incorporated in the Province of Ontario pursuant to the Business Corporations Act (Ontario) by filing Articles of Incorporation effective June 3, 1996.

The executive head office, registered office and principal place of business of the issuer are located in the city of Greater Sudbury at 129 Fielding Road, Lively, Ontario, P3Y 1L7. The issuer also maintains an office at 80 Richmond Street West, 18th Floor, Toronto, Ontario, M5H 2A4.

The issuer's common shares are listed on the Toronto Stock Exchange (TSX) under the symbol "WM".

2.2 Terms of Reference

Wallbridge's acquisition of the Fenelon Mine Property from Balmoral Resources Ltd ("Balmoral") commenced in May 2016 and the purchase was completed in October 2016 (Wallbridge press releases of May 25, 2016 and October 19, 2016). The "Fenelon Mine Property" of Wallbridge corresponds to the former "Discovery Zone Property" of Balmoral. At the time of its acquisition, the area covered by the Fenelon Mine Property represented a 10.5-km^2 subdivision of the larger Fenelon Property owned by Balmoral (Fig. 2.1). Balmoral's Fenelon Property has also been called the "Fenelon A Property" or the "Fenelon Project" by past operators. The gold deposit on the Fenelon Mine Property is currently known as the "Fenelon deposit" or the "Fenelon gold mine" by the issuer, but was formerly known as the "Discovery gold zone" or "Discovery Zone deposit" by Balmoral. The terms are considered synonymous in this report.

The Fenelon Mine Property is an advanced stage project with near-term production potential, and drill intersections suggest an exploration potential for resource expansion. The Property is situated near the Sunday Lake Deformation Zone, which hosts the Detour Lake mine in Ontario (Detour Gold Corporation) and the Martiniere gold project in Québec (Balmoral).

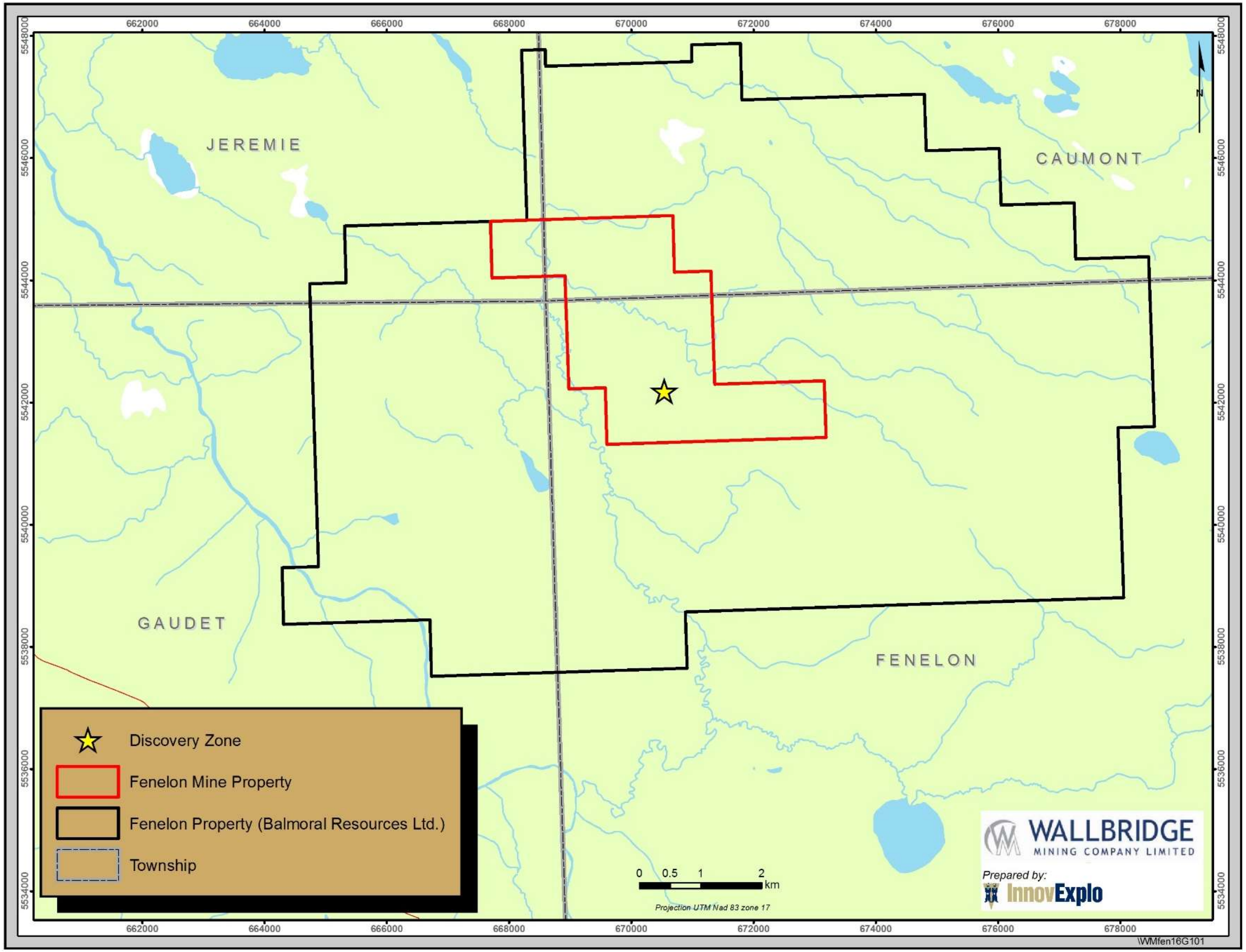


Figure 2.1 – Location of Wallbridge's Fenelon Mine Property and the Fenelon Property owned by Balmoral

The Discovery Zone was discovered in 1994. In all, more than 50,000 metres have been drilled, and importantly, two bulk samples have been mined and processed from the deposit. In 2001, a 13,835-tonne bulk sample mined from a small open pit at the Discovery Zone was test-milled at the Camflo Mill in Malartic, returning 132,039 grams (4,245 oz) of gold for a reconciled grade of 9.84 g/t Au using a calculated recovery of 97%. In 2004, a second bulk sample, mined from underground and also milled at Camflo, consisted of 8,169 tonnes and returned 80,731 grams (2,596 oz) of gold for a reconciled grade of 10.7 g/t Au. The open pit and underground workings are currently flooded.

This Technical Report was prepared by InnovExplo to present and support the PFS for the Fenelon Mine Property. Wallbridge announced the positive results in a press release on February 2, 2017. The 2017 PFS estimates a pre-tax net cash flow of $6.62 million and a pre-tax internal rate of return ("IRR") of 92% for the initial approximately 18-month mine life for the reserves contained in the uppermost 100 metres of the deposit and in close proximity to the existing ramp.

2.3 Principal Sources of Information

As part of the current mandate, the qualified and independent persons (QPs) for the report, as defined by NI 43-101, have reviewed the following aspects with respect to the Fenelon Mine Property: mining titles and their status recorded in GESTIM (the Government of Québec's online claim management system); agreements and technical data supplied by the issuer (or its agents); public sources of relevant technical information available through SIGÉOM (the Government of Québec's online warehouse for assessment work); and the issuer's filings on SEDAR (press releases and Management's Discussion & Analysis reports).

Some of the geological and/or technical reports for the Fenelon Mine Property or projects in its vicinity were prepared before the implementation of NI 43-101 in 2001. The authors of such reports appear to have been qualified and the information prepared according to standards that were acceptable to the exploration community at the time. In some cases, however, the data are incomplete and do not fully meet the current requirements of NI 43-101. InnovExplo has no known reason to believe that any of the information used to prepare this Technical Report is invalid or contains misrepresentations. The authors have sourced the information for the Technical Report from the collection of reports listed in Item 27 – *References*.

InnovExplo believes the information used to prepare the Technical Report and to formulate its conclusions and recommendations is valid and appropriate considering the status of the project and the purpose for which the report is prepared. The authors, by virtue of their technical review of the project, affirm that the work program and recommendations presented in the report are in accordance with NI 43-101 and CIM Definition Standards for Mineral Resources and Mineral Reserves.

The QPs do not have, nor have they previously had, any material interest in the issuer or its related entities. The relationship with the issuer is solely a professional association between the issuer and the independent consultants. The Technical Report was prepared in return for fees based upon agreed commercial rates, and the payment of these fees is in no way contingent on the results of the Technical Report.

2.4 Qualified Persons

The qualified and independent persons ("QPs") for the Technical Report are as follows:

- Pierre-Luc Richard, P.Geo. (OGQ #1119), Director Geology (InnovExplo).
- Bruno Turcotte, P.Geo. (OGQ #453), Senior Geologist (InnovExplo).
- Catherine Jalbert, P.Geo. (OGQ #1412), Geologist (InnovExplo).
- Denis Gourde, P.Eng. (OIQ # 43860), Vice-President – Engineering and Sustainable Development (InnovExplo).
- Pierre Pelletier, P.Eng. (OIQ #36825), Technical Advisor, Metallurgy (InnovExplo).
- George Darling, P. Eng. (PEO # 1049701), Mining Engineer (SNC-Lavalin).
- Marie-Claude Dion St-Pierre, P.Eng. (OIQ #40947) Project Director (WSP Canada.).

The list below presents the sections of the Technical Report for which each QP was responsible:

- Pierre-Luc Richard: co-author of sections 1, 14, and 25 to 27.
- Bruno Turcotte: author of sections 2 to 11, 23 and 24; co-author of sections 1, 25 to 27.
- Catherine Jalbert: author of section 12; co-author of sections 1, 14, and 25 to 27.
- Denis Gourde: author of section 19.
- George Darling: author of sections 15, 16, 18, 21 and 22; co-author of sections 1 and 25 to 27.
- Marie-Claude Dion St-Pierre: author of section 20; co-author of sections 1 and 25 to 27.
- Pierre Pelletier: author of sections 13 and 17; co-author of sections 1, 25 and 26.

2.5 Inspection of the Property

George Darling, P.Eng., visited the Fenelon Mine Property on October 12, 2016. Mr. Darling was accompanied by Marz Kord, P.Eng., of Wallbridge.

Catherine Jalbert, P.Geo., B.Sc., also visited the Fenelon Mine Property on May 31 and June 1, 2016, during the course of a previous mandate (2016 Mineral Resource Estimate). Ms. Jalbert was accompanied by Attila Pentek, P.Geo., of Wallbridge.

2.6 Effective Date

The effective date of the Technical Report is February 2, 2017.

2.7 Abbreviations, Units and Currencies

A list of abbreviations used in this report is provided in Table 2.1. All currency amounts are stated in Canadian Dollars ($, $C, CAD) or US dollars ($US, USD). Quantities are stated in metric units, as per standard Canadian and international practice, including metric tons (tonnes, t) and kilograms (kg) for weight, kilometres (km) or metres (m) for distance, hectares (ha) for area, percentage (%) for copper and nickel grades, and gram per metric ton (g/t) for gold, platinum and palladium grades. Wherever applicable, imperial units have been converted to the International System of Units (SI units) for consistency (Table 2.1).

Table 2.1 – List of abbreviations

Unit or Term	Abbreviation or Symbol
acid rock drainage	ARD
American dollars	US$ or USD
billion	G
billion years	Ga
Bureau d'audience publique du Québec	BAPE
Canadian dollar	$, C$, CAD
Canadian Environmental Assessment Agency	CEAA
certificate of authorization	CA
centimetre	cm
cfm	cubic feet per minute
chalcopyrite	cpy
CIP	carbon-in-pulp
cobalt	Co
copper	Cu
cubic metre	m^3
decametre	dm
degree Celsius	°C
diamond drill hole	DDH
Directive 019 sur l'industrie minière	Directive 019
electromagnetic	EM
environmental impact assessment	EIA
environmental and social impact assessment	ESIA
exchange rate	CAD:USD
feasibility study	FS
foot	ft, '
general and administration	G&A
gigawatt	GW
gold	Au
gold equivalent	AuEq
gram	g
gram per cubic centimetre	g/cm^3
gram per metric ton	g/t
hectare	ha
horizontal loop electromagnetic	HLEM
hour	hr, h
inch	in, "

Unit or Term	Abbreviation or Symbol
induced polarization	IP
inductively coupled plasma	ICP
iron	Fe
joint venture	JV
kilogram	kg
kilometre	km
kilowatt	kW
kilowatt-hour	kWh
life of mine	LOM
life of mine plan	LOMP
magnetometer, magnetometric	Mag
megawatt	MW
Metal Mining Effluent Regulations	MMER
metre	m
metres above sea level	masl
metric ton (tonne)	t
micron (micrometre)	µm
millimetre	mm
million	M
million metric tons	Mt
million ounces	Moz
million years	Ma
minute	min
Ministère de l'Énergie et des Ressources Naturelles du Québec	MERN
Ministère des Forêts, de la Faune et des Parcs	MFFP
Ministère du Développement durable, de l'Environnement et de la Lutte contre les changements climatiques	MDDELCC
National Instrument 43-101	NI 43-101, 43-101
net present value	NPV
net smelter return	NSR
nickel	Ni
nickel equivalent, nickel equivalent pounds	NiEq, lbs NiEq
ounce per short ton	oz/st, oz/t
palladium	Pd
part per billion	ppb
part per million	ppm
platinum	Pt

Unit or Term	Abbreviation or Symbol
platinum group elements	PGE
platinum group metals	PGM
potential acid generation	PAG
pyrite	py
pyrrhotite	po
run of mine	ROM
semi-autogenous-grinding	SAG
short ton	st, ton
silver	Ag
thousand	k
thousand ounces	koz
tonne	t
tonnes (metric tons) per day	tpd
troy ounce	oz
troy ounce	oz
tungsten	W
versatile time domain electromagnetic	VTEM
volcanogenic massive sulphide	VMS
zinc	Zn

Table 2.2 – Conversion factors for measurements

Imperial Unit	Multiplied by	Metric Unit
1 inch	25.4	mm
1 foot	0.3048	m
1 acre	0.405	ha
1 ounce (troy)	31.1035	g
1 pound (avdp)	0.4535	kg
1 ton (short)	0.9072	t
1 ounce (troy) / ton (short)	34.2857	g/t

3 RELIANCE ON OTHER EXPERTS

- The QPs relied on the following sources for information outside their field of expertise or beyond the scope of the current mandate The issuer supplied information about mining titles, option agreements, royalty agreements, environmental liabilities and permits. Neither the QPs nor InnovExplo are qualified to express any legal opinion with respect to property titles or current ownership and possible litigation. This disclaimer applies to sections 4.2 to 4.7 of this report.
- Sylvie Poirier, P.Eng., and Denis Gourde, P.Eng., both of InnovExplo, supplied the 2016 MRE cut-off grade parameters of the previous technical report on the Fenelon Mine property.
- Venetia Bodycomb, M.Sc., of Vee Geoservices edited a draft version of this report.

The authors believe the information used to prepare the report and to formulate its conclusions and recommendations is valid and appropriate considering the status of the project and the purpose for which the report is prepared.

The authors, by virtue of their technical review of the project’s exploration potential, affirm that the work program and recommendations presented in the report are in accordance with NI 43-101 and CIM technical standards.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Fenelon Property is located in the Nord-du-Québec administrative region in the province of Québec (Canada), approximately 75 kilometres west-northwest of the city of Matagami (Fig. 4.1).

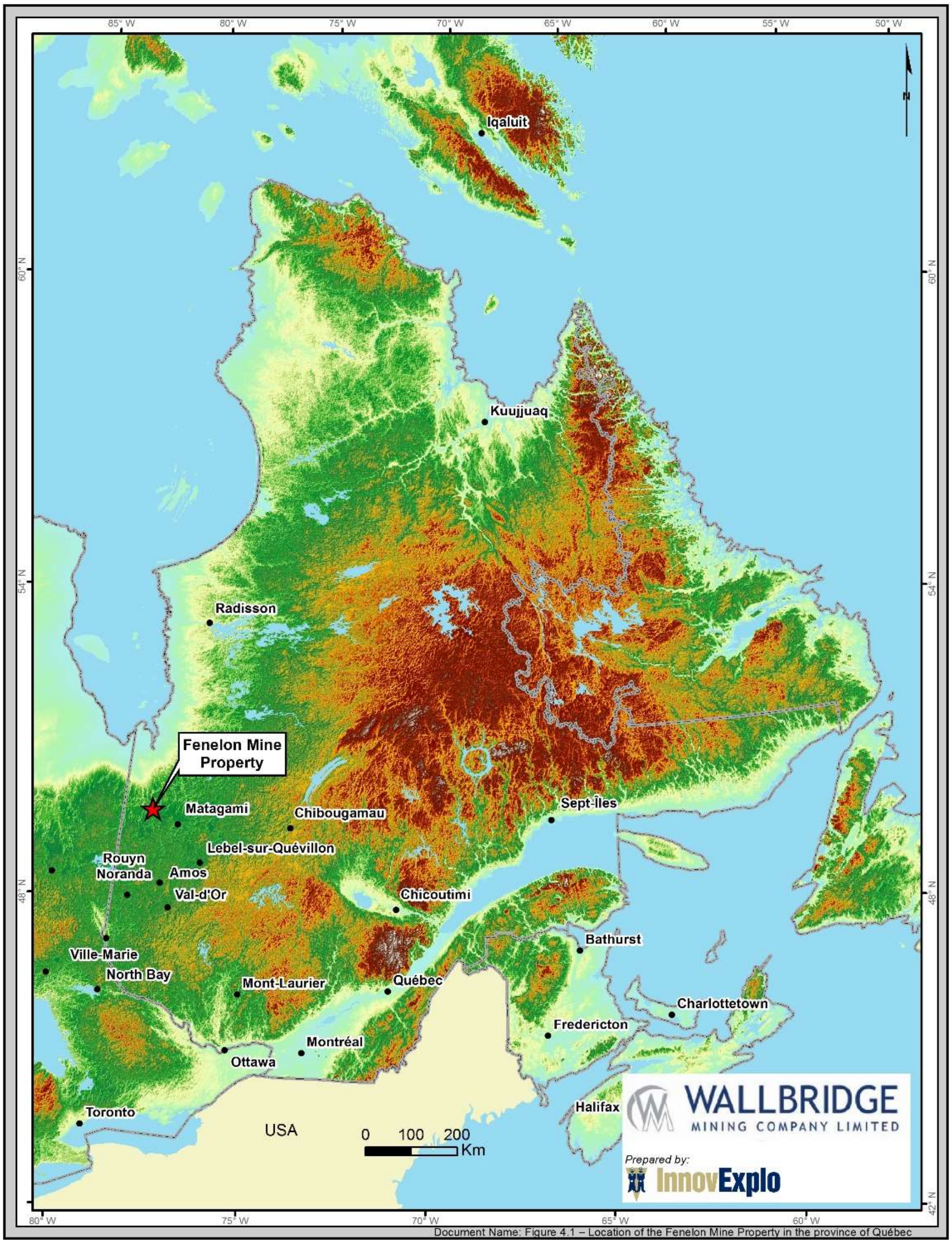


Figure 4.1 – Location of the Fenelon Mine Property in the province of Québec

The approximate centroid of the Fenelon Mine Property is 78°37'30"W and 50°01'00"N (UTM coordinates: 670140E and 5543175N, NAD 83, Zone 18). The nearest community is Matagami, located about 75 kilometres east-southeast of the Property. The Property lies in the townships of Fenelon, Caumont and Jérémie on NTS maps sheet 32L/02.

4.1.1 Mining Rights in the Province of Québec

The following discussion on mining rights in the province of Québec was mostly summarized from Guzun (2012), Gagné and Masson (2013) and the *Act to Amend the Mining Act* (Bill 70) assented on December 10, 2013 (National Assembly, 2013). The reader is referred to Appendix I for a detailed discussion on mining rights in the province of Québec.

In Québec, mining and mineral exploration is principally regulated by the provincial government. The *Ministère de l'Énergie et des Ressources Naturelles du Québec* ("MERN"; a.k.a. Ministry of Energy and Natural Resources) is the provincial agency entrusted with the management of mineral substances in Québec. The ownership and granting of mining titles for mineral substances are primarily governed by the *Mining Act*, as amended by Bill 70, and its attending regulations. In Québec, land surface rights are distinct property from mining rights. Rights in or over mineral substances in Québec form part of the domain of the State (the public domain), subject to limited exceptions for privately owned mineral substances. Mining titles for mineral substances within the public domain are granted and managed by the MERN. The granting of mining rights for privately owned mineral substances is a matter of private negotiations, although certain aspects of the exploration for and mining of such mineral substances are governed by the *Mining Act*.

4.1.2 The Claim

The claim is the only exploration title currently issued in Québec for mineral substances (other than surface mineral substances, petroleum, natural gas and brine). A claim gives its holder the exclusive right to explore for such mineral substances on the land subject to the claim, but does not entitle its holder to extract mineral substances, except for sampling and only in limited quantities. In order to mine mineral substances, the holder of a claim must obtain a mining lease. Electronic map designation is the most common method of acquiring new claims from the MERN, whereby an applicant makes an online selection of available pre-mapped claims. There are only a few places in the province where claims can still be obtained by staking.

4.1.3 The Mining Lease

Mining leases are extraction (production) mining titles that give their holder the exclusive right to mine mineral substances (other than surface mineral substances, petroleum, natural gas and brine). A mining lease is granted to the holder of one or several claims upon proof of the existence of indicators of the presence of a workable deposit on the area covered by such claims and compliance with other requirements prescribed by the *Mining Act*. A mining lease has an initial term of 20 years, but may be renewed for three additional periods of 10 years each. Under certain conditions, a mining lease may be renewed beyond the three statutory renewal periods.

4.1.4 The Mining Concession

Mining concessions are extraction (production) mining titles that give their holder the exclusive right to mine mineral substances (other than surface mineral substances, petroleum, natural gas and brine).

Mining concessions were issued prior to January 1, 1966. After that date, grants of mining concessions were replaced by grants of mining leases. Although similar in certain respects to mining leases, mining concessions granted broader surface and mining rights and are not limited in time. A grantee must commence mining operations within five years from December 10, 2013. As is the case for a holder of a mining lease, a grantee may be required by the government, on reasonable grounds, to maximize the economic spinoffs within Québec of mining the mineral resources authorized under the concession. The grantee must also, within three years of commencing mining operations and every 20 years thereafter, send the Minister a scoping and market study as regards to processing in Québec.

4.2 Mining Title Status

Mining title status for the Fenelon Mine Property was supplied by Marz Kord, President and CEO for Wallbridge. InnovExplo verified the status of all mining titles using GESTIM, the Québec government's online claim management system (http://gestim.mines.gouv.qc.ca; via Internet Explorer browser only).

The Fenelon Mine Property currently consists of one block of nineteen (19) mining claims staked by electronic map designation ("map-designated cells") and one (1) mining lease, for an aggregate area of 1,051.77 ha (10.5 km^2; Fig. 4.2). All mining titles are registered 100% in the name of Wallbridge Mining Company Ltd. All mining titles are in good standing according to the GESTIM database. A detailed list of mining titles, ownership, royalties and expiration dates is provided in Appendix II.

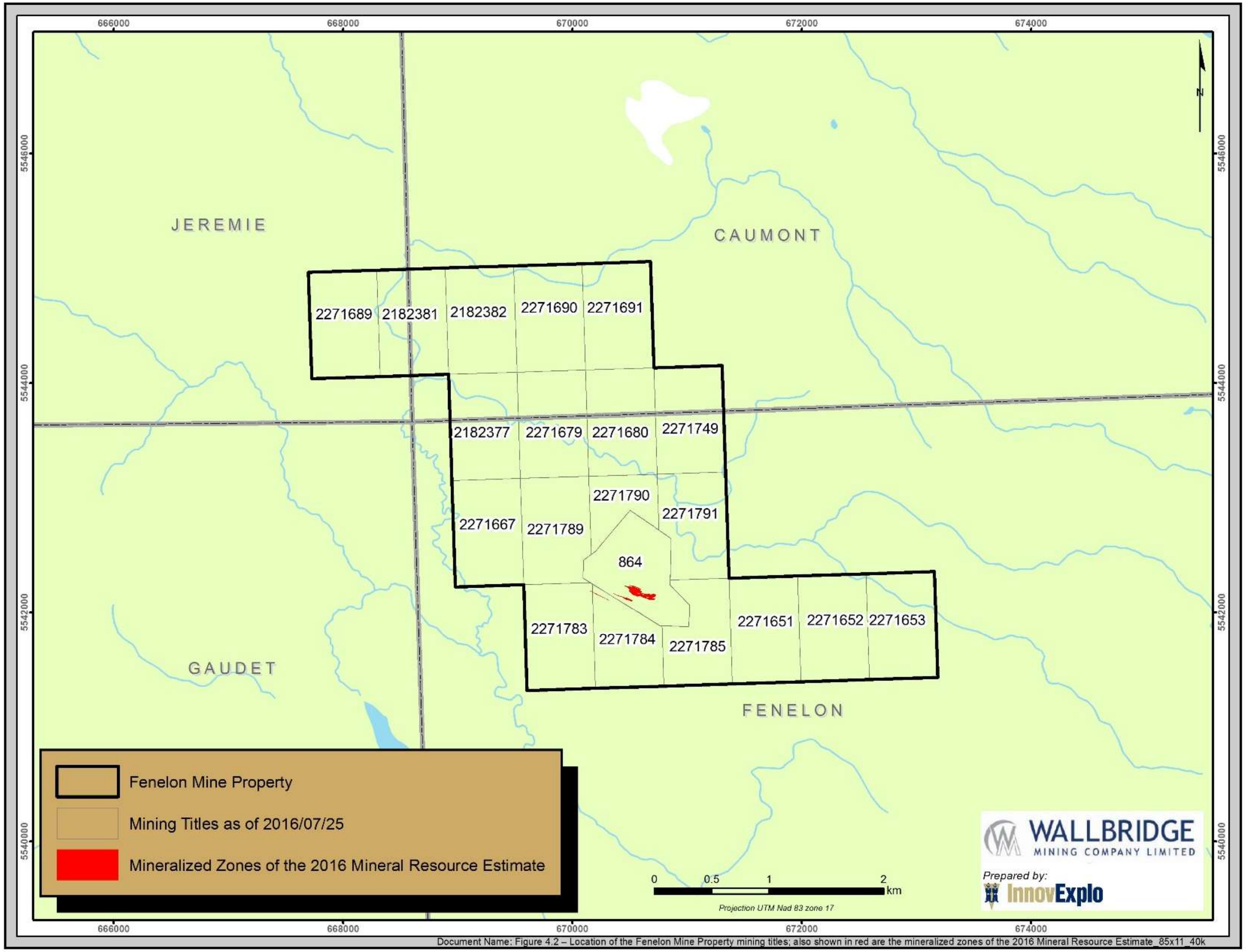


Figure 4.2 – Location of the Fenelon Mine Property mining titles and the mineralized zones covered by the 2016 Fenelon Deposit Mineral Resource Estimate.

4.3 Acquisition of the Fenelon Mine Property

On May 25, 2016, Wallbridge announced it had entered into a binding Letter of Intent (“LOI”) dated May 24, 2016 (the “LOI Date”) to acquire a 100% interest in a 10.5-km^2 subdivision of the Fenelon Property from Balmoral Resources Ltd for a purchase price of C$3.6 million.

The LOI outlined the terms of the proposed transaction, which are as follows:

- Wallbridge shall immediately upon receipt of TSX approval, issue to Balmoral that number of common shares in the capital of Wallbridge as is equal to C$200,000 based on the 20-day volume weighted average trading price of Wallbridge's common shares in the 20 days immediately prior to market close on May 20, 2016, said payment equalling 2,381,575 common shares of Wallbridge. The shares issued will be subject to standard four-month hold provisions.
- The parties shall, using their respective best efforts, prepare a purchase agreement (the “Purchase Agreement”) to confirm and expand on the terms outlined in the LOI. It is the intention of the parties that the Purchase Agreement shall be signed within 60 days of the LOI Date.
- Under the terms of the LOI, the purchase price for the Property, if paid by Wallbridge to Balmoral within 60 days of LOI Date, will be C$3,400,000 cash.
- Should Wallbridge not be in a position to make the required cash payment within 60 days of the LOI Date, the cash purchase price will increase to C$3,500,000. Wallbridge may extend the final deadline for payment to 120 days from the LOI Date by making two non-refundable cash payments to Balmoral of C$500,000 each on or before the 60th and 90th day from the LOI Date. Both payments will form part of the final purchase price.
- Should the Purchase Agreement not be completed and/or the purchase payment(s) not be received by Balmoral under the terms outlined above, then the LOI and/or the Purchase Agreement (if completed) shall automatically terminate. Upon termination of the LOI and/or Purchase Agreement, Wallbridge will retain no interest in the Property and Balmoral will be entitled to retain any payments previously received under the terms of the LOI and/or Purchase Agreement.
- In all cases, Balmoral shall retain a 1% NSR on any future production from the Property.

In the press release of October 19, 2016, Wallbridge announced it had completed the purchase of the Fenelon Mine Property by making the final payment of $2,500,000 towards the purchase price. The Property is subject to the 1% NSR in favour of Balmoral, as well as other previous encumbrances as outline in the following section.

4.4 Previous agreements and encumbrances

The following relevant paragraph was taken from the 2010 technical report by Leclerc and Giguère (2010). It was prepared by Cory H. Kent, legal counsel to American Bonanza Gold Corp. (“Bonanza”), and it outlines the existing royalty obligations on the Fenelon Property as it was defined at the time:

"Pursuant to an agreement dated July 17, 1998, as amended May 1, 2000, between Cyprus Canada Inc. (now owned by Freeport McMoRan Copper and Gold Inc.) and International Taurus Resources Inc. (a predecessor company to American Bonanza Gold Corp.), American Bonanza Gold Corp. (the "Option Agreement") has the right to explore and acquire all of Cyprus interest in Cyprus' entire Casa Berardi exploration portfolio in the province of Québec, Canada (the Casa Berardi Properties). The Casa Berardi Properties consist of four properties: the Fenelon Project, Martiniere "D", Northway and La Peltrie located within the Casa Berardi sector of the Abitibi Greenstone Belt. Pursuant to the Option Agreement, in order to acquire the remaining interests in the Casa Berardi Properties, Bonanza is required to pay three installments of US$150,000 (total US$450,000), with the first installment to be paid upon commencement of commercial production on any one of the properties and the remaining installments to be made six and twelve months thereafter. Cyprus will maintain a net smelter return royalty to a maximum of 2% (on properties not having an underlying royalty burden) and minimum of 1% (on those properties having an underlying royalty) on commercial production from the Casa Berardi Properties. The Corporation acquired its 38% interest in the Fenelon project and an option to acquire the remaining 62% in accordance with the Option Agreement as a result of its merger with International Taurus Resources in 2005."

Under the terms of a purchase and sale agreement dated November 3, 2010 ("Bonanza Agreement") and completed November 9, 2010, Balmoral purchased Bonanza's rights to and interests in the Fenelon, N2, Martiniere and Northshore properties, along with certain surface rights attached to the Northshore Property, an existing exploration camp and materials at the Fenelon Property and property-related exploration data. Balmoral acquired a significant interest and operational control in each of the properties and the right to acquire a 100% interest, subject to certain royalty interests, in each of the properties upon payment of US$450,000 to Cyprus Canada on or before the commencement of commercial production from any of the properties. In consideration for the acquisition of the foregoing assets from Bonanza, Balmoral paid C$3,700,000 and issued 4,500,000 common shares to Bonanza. The shares were sold subsequently on the open market.

Balmoral acquired from Bonanza its current 38% undivided interest in the Fenelon Property along with the Option ("Cyprus Option") to purchase the remaining 62% interest in the property from Cyprus Canada Ltd. (now Freeport McMoRan Copper and Gold Inc.). According to the terms, Balmoral could exercise the Cyprus Option and vest a 100% interest in the Fenelon Property, subject only to the royalty interest described below, by making an additional one-time payment of US$450,000 in favour of Cyprus Canada, said payment being due on commencement of commercial production from the Fenelon Property or the other properties to be acquired by Balmoral from Bonanza. Upon making the required payments, Balmoral would hold a 100% interest in the property subject only to a 1% NSR in favour of Cyprus Canada and annual claim holding costs.

In January 2013, Balmoral completed the acquisition of a 100% interest in the Fenelon Property from Cyprus Canada and granted a 1% NSR on the property in favour of Cyprus Canada as required by the acquisition agreement.

As a result of these previous agreements, the Fenelon Mine Property is subject to a net smelter royalty (NSR) of 1% payable from production on the property to Cyprus Canada Ltd, and an NSR royalty of 1% payable from production on the property to Balmoral Resources Ltd.

4.5 Access to the Property

The Fenelon Mine Property is situated on Crown land in the Eeyou Istchee–James Bay Territory. It is subject to the James Bay and Northern Québec Agreement (JBNQA) and falls under Category III lands as defined by that agreement. Mineral exploration is allowed under specific conditions.

The JBNQA Environmental Protection Regime covers the protection of Native hunting, fishing and trapping rights. Category III lands are public lands on which Native people can carry on their traditional activities year-round, and on which they have exclusive rights to certain animal species. Each hunting area has a tallyman.

The issuer should communicate with the regional level of government and the Cree Nation Government on these matters.

4.6 Permits

Permits are required for any exploration program that involves tree cutting to provide road access for the drill rig or to carry out drilling and stripping work. Permitting timelines are short, typically about 3 to 4 weeks. The permits are issued by the *Ministère des Forêts, de la Faune et des Parcs* (MFFP; Ministry of Forestry, Wildlife and Parks).

4.7 Environment

There are no environmental liabilities pertaining to the Fenelon Mine Property.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The main access to the Fenelon Mine Property (Fig. 5.1) is via Highway 109 from Amos, which heads north to Matagami and Radisson. At the junction with the road leading to the former small mining town of Joutel, head west for just over 13 kilometres, then turn northwest on the Selbaie paved road (N-810) for a distance of 51 kilometres. Past the bridge over the Harricana River (at Km 122) and just short of the kilometre 123 marker, the Tembec forestry provides access to Balmoral's Fenelon Camp at a distance of 21 kilometres from the junction. The old open pit and decline ramp are located 6 kilometres west of the Fenelon Camp.

5.2 Climate

The region experiences a typical continental-style climate, with cold winters and warm summers. Climate data from the nearest weather station in the town of Matagami, Québec, indicate that daily average temperatures range from -20 °C in January to 16 °C in July (Environment Canada, 2012). The coldest months are December to March, during which temperatures are often below -30 °C and can fall below -40 °C. During summer, temperatures can exceed 30 °C. Snow accumulation begins in October or November and the snow cover generally remains until the spring thaw in mid-March to May. The average monthly snowfall peaks at 65 cm in February and the yearly average is 314 centimetres (Environment Canada, 2012). Drilling can be conducted year-round, with the exception of the spring thaw period from mid-March to May.

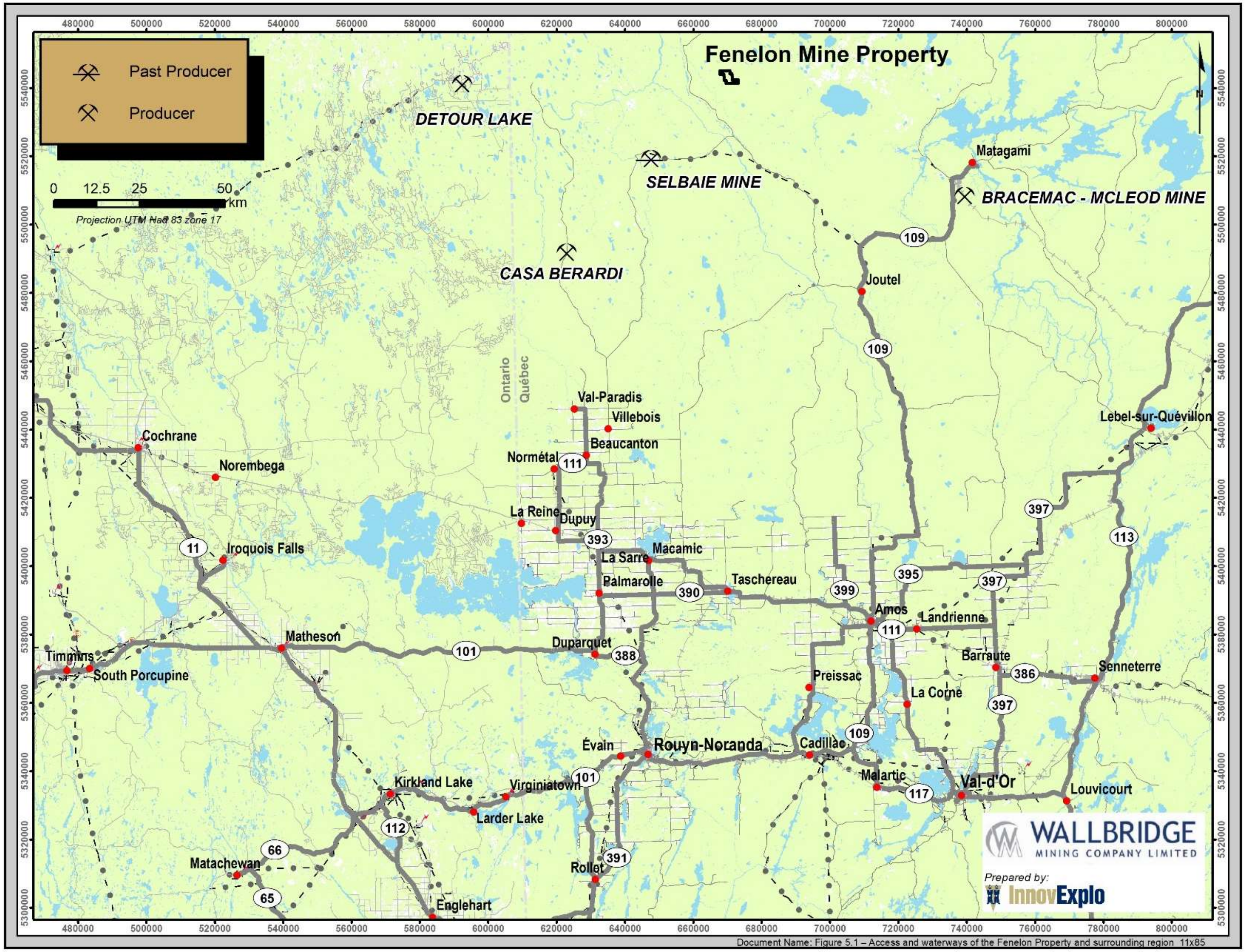


Figure 5.1 – Access and waterways of the Fenelon Mine Property and surrounding region

5.3 Local Resources

The Fenelon Mine Property can obtain supplies, personnel and maintenance support from the nearby towns of Amos (pop. 12,671) and Val-d'Or (pop. 31,862), (Statistics Canada, 2011). Both Amos and Val-d’Or have road access to the Property and offer a full range of mineral exploration services and supplies. A number of mining and mineral exploration companies have offices in Val-d’Or. Local available resources include the following:

- Assayers – commercial laboratories (Val-d’Or);
- Civil construction companies (Amos and Val-d’Or);
- Diamond drilling – multiple contractors (Amos and Val-d’Or);
- Engineering firms (Val-d’Or);
- Freight forwarding (Amos and Val-d’Or);
- Geological consultants (Val-d’Or);
- Geophysics contractors (Val-d’Or);
- Land surveyors (Amos and Val-d’Or);
- Mining contractors (Val-d’Or); and
- Suppliers of industrial mining equipment, including diesel engines, explosives, mechanical parts, electrical supplies and cable, electronics and tires (Amos and Val-d’Or).

The nearest helicopter bases are in Cochrane (Ontario) and La Sarre (Québec), respectively located 210 kilometres southwest and 140 kilometres south of the Fenelon Mine Property. Val-d’Or has the nearest regional airport, with daily flights to various destinations. The nearest rail access is the CN Rail line to Matagami, about 75 kilometres east-southeast of the Fenelon Mine Property.

5.4 Infrastructure

No high voltage power line is available on or near the Fenelon Mine Property. There is an ample supply of water on or near the property to supply a mining operation. An old garage (Fig. 5.2) is still present near the flooded open pit (Fig. 5.3).

Accommodations at Balmoral’s Fenelon Camp (Fig. 5.4) consist of ATCO trailers with indoor plumbing, a potable water well and forced-air heating. Electricity runs on a 78-kW generator. The camp has the capacity to support up to 25 people.

Figure 5.2 – Flooded open pit on the Fenelon Mine Property (from Balmoral's website).

Figure 5.3 – Old garage used during the 2004 mining operations (photo taken during May 2016 site visit).

Figure 5.4 – Access road, typical physiography of the area, and Balmoral's Fenelon Camp

5.5 Physiography

The Fenelon Mine Property has a thick and extensive cover of Pleistocene glacial sediments ranging from 50 to 100 metres thick. Bedrock exposures are scarce, locally occurring on small knolls and along major rivers. The low parts of the Property are almost devoid of outcrops. Most of the area is covered with swamps and flat forests formed by spruce, fir and pine (Fig. 5.4). Some areas of the Property have recently been logged and partly revegetated. The minimum and maximum elevations on the property are 250 masl and 320 masl, respectively.

6 HISTORY

6.1 1980–1982 Exploration Program (Teck Explorations)

The area covered by the current Fenelon Mine Property was covered by a DIGHEM survey by Teck Explorations Ltd. Following this survey, three anomalies in the southeast part of the Property were selected and staked in the field. These anomalies were situated. Between February and March 1981 and in March and April 1982, Teck carried out ground Pulse EM, MaxMin II HLEM and Mag surveys over these anomalies (Thorsen 1981a; 1981b; 1982a).

6.2 1986–1991 Exploration Program (Morrison–Total Energold)

Between August 14 and December 20, 1986, the area covered by the current Fenelon Mine Property was surveyed by Aerodat Ltd for parent company Morrison Minerals Limited ("Morrison"), in turn a wholly owned subsidiary of Morrison Petroleums. The combined helicopter-borne magnetic and electromagnetic survey included a three-frequency electromagnetic system, a cesium high sensitivity magnetometer, a two frequency VLF-EM system, a tracking camera and a radar positioning system (Boustead, 1988). The flight lines were oriented at an azimuth of N360° and a spacing of 100 metres. The survey was flown at a mean clearance of 60 metres.

In February 1989, Morrison carried out a ground HEM and magnetic surveys on their Fenelon "A" Property, covering about half of what would later become Balmoral's Fenelon Property (Turcotte and Gauthier, 1989). At the time, the Fenelon "A" property consisted of fourteen (14) staked claims.

In 1990, a joint venture agreement (Casa Berardi Joint Venture: "CBJV") was signed between Total Energold Corporation ("Total Energold") and Morrison, allowing the partners to pursue exploration targets in the Casa Berardi area (including the current Fenelon Mine Property area), using all geophysical data and an overlying AutoCAD compilation.

In January 1991, Morrison and Total Energold staked twenty-four (24) claims adjoining their Fenelon "A" Property for a total of 38 staked claims. In late January and early February of 1991, geophysical surveys were carried out to locate and better define target areas selected from earlier airborne survey data (Kenwood, 1991). Ground MaxMin II and total field magnetic surveys were then conducted. The magnetics survey covered 16.1 line-kilometres with stations every 25 metres. A strong HLEM conductor was identified along the flank of a strong magnetic high in the central part of the survey. The weaker shallow conductors at the northwest end of the survey were also associated with strong magnetic axes.

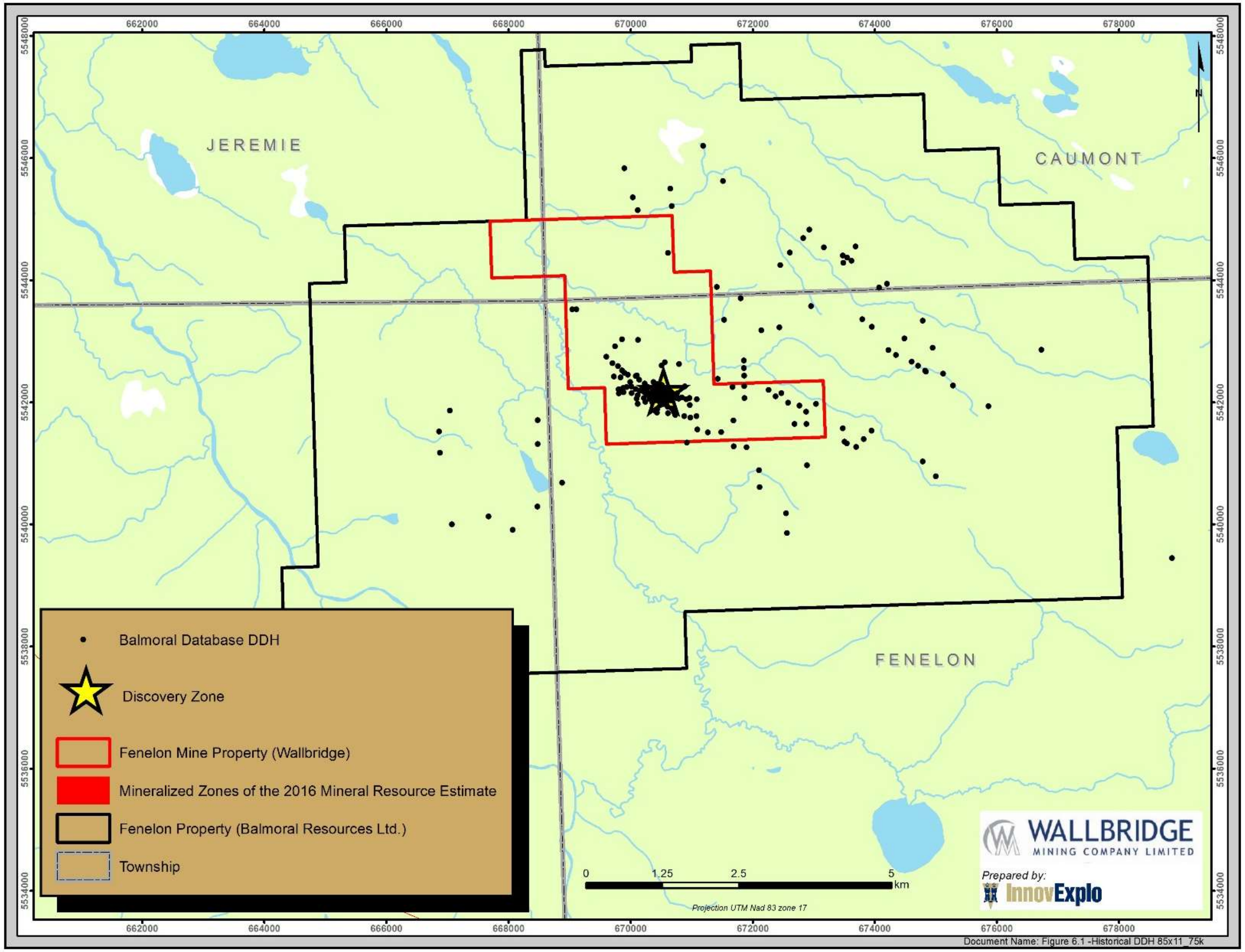


Figure 6.1 – Map showing Wallbridge's Fenelon Mine Property surrounded by Balmoral's Fenelon Property.
(Note: only the DDH on the Fenelon Mine Property were validated as part of this study)

6.3 1992–1993 Exploration Program (Cyprus-OGY)

On October 1992, Cyprus Canada Inc. ("Cyprus") purchased the original CBJV interest of Total Energold Corporation that included the Fenelon "A" Property. In November 1992, Morrison Minerals Limited was amalgamated with OGY Petroleums Ltd ("OGY"). At this time, Cyprus had the possibility to earn a 55% interest in the joint venture with OGY (45%). Cyprus was the operator of the CBJV.

During the winter 1993, Cyprus drilled the first hole on the Fenelon "A" Property. Only sixteen (16) staked claims of the original thirty-eight (38) had been maintained with assessment credits prior to the 1993 drill programs (Broughton, 1993).

In February 1993, the BQ-caliber hole FA93-1, totalling 185 metres, tested an HLEM conductor striking N125° across the eastern part of the property. This magnetic feature could be traced southeastward to Teck drill hole GB-68-1 (580 ppb Au over 0.51 m; see Thorsen, 1982b). Hole FA93-1 was collared at the strongest part of the Mag high, coincident with the best response from the flanking conductor, approximately 1,200 metres along the strike from the Teck drill hole.

The hole intersected a 35-metre-wide sericite-chlorite-Fe-carbonate alteration zone centred on a sequence of locally pyritic interbedded sediments, iron formations and volcanics, intruded by feldspar porphyry dykes. A pyritic-chloritic iron formation at the top of the sequence returned 2.84 g/t Au over 0.95 metre, and the pyritic sediments were anomalous in gold throughout. The alteration zone was also anomalous in arsenic (up to 1,800 ppm As), copper (up to 537 ppm Cu) and zinc (up to 3,840 ppm Zn).

6.4 1994 Exploration Program (Cyprus-OGY)

Between February and April 1994, Cyprus added 1,425.8 metres of drilling in eight (8) BQ-size holes (FA94-2 to FA94-9) on the Fenelon "A" property (Guy, 1994). The drilling program was initiated to follow up on alteration and mineralization intersected in 1993 (hole FA93-1), which indicated the presence of hydrothermal alteration and a geological environment favourable for gold mineralization. Hole FA94-2 was drilled southeast of the 1993 drill hole, between hole FA93-1 and Teck hole GB-68-1. The intersected geology was similar to the hole FA93-1. No significant gold values were obtained.

Hole FA94-3 was located 1,300 metres southwest of hole FA93-1. The hole was targeted on a proposed volcanic/sediment contact with coincident conductivity, flanking a magnetic high in the vicinity of a set of northeast trending faults. The hole was drilled entirely in sediments with the conductivity explained as graphitic argillite with massive pyrite, and the magnetic anomaly explained as pyrrhotite mineralization in greywacke and argillite.

Hole FA94-4 (Fig. 6.2) was collared 1,000 metres northwest of FA94-3. The target was a magnetic feature that appeared to represent a flexure or fold in the volcanic stratigraphy. Geophysically, the target was a conductive zone flanking a magnetic high interpreted as a mineralized alteration zone. The hole was collared in sediments and progressed into a fine-grained mafic to ultramafic intrusive. Within this intrusive, two silicified sections were observed with pyrrhotite, chalcopyrite and visible gold.

These sections assayed 42.6 g/t Au over 6.7 metres (uncut), including 144.5 g/t Au over 2.1 metres (uncut). This represents the discovery hole for the Discovery Zone. Alteration surrounding the mineralized intercept consists of purple-brown biotite and iron carbonate. The gold intercept was anomalous in copper with values in the range of 0.2% to 1% Cu. The remainder of the hole mainly intersected a sequence of sediments with quartz-feldspar porphyry dykes.

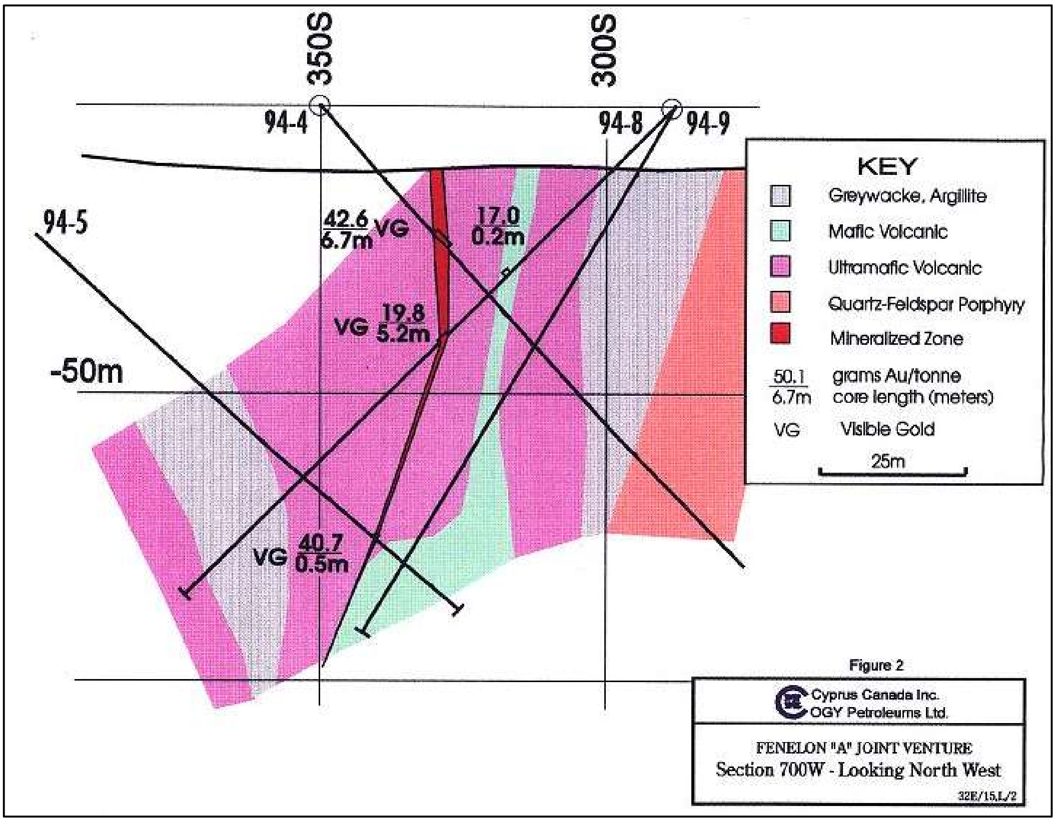


Figure 6.2 – Cross section 700W showing the discovery hole FA94-4 (from Guy, 1994)

Holes FA94-5 to FA94-9 were drilled as a follow-up to hole FA-94-4. Holes FA94-5, FA94-8 and FA-94-9 were drilled on the same section and below hole FA94-4 (Fig. 6.2). Hole FA94-5 represented the deepest intercept on the section at the -75 metre elevation. It intersected a 2.3-metre silicified zone at the contact between ultramafic units and mafic volcanic flows. A silicified 0.5-metre section assayed 40.73 g/t Au.

Hole FA94-8 was drilled between hole FA94-4 and FA94-5 to intersect the mineralized zone at the -40 metre elevation (Fig. 6.2). The hole intersected an ultramafic, hosted quartz vein system with visible gold that assayed 19.8 g/t Au over 5.2 metres. Hole FA94-9 was drilled beneath of hole FA94-8 to test the possibility that the quartz vein system was located ahead of hole FA94-5 (i.e., it had not been drilled far enough). Hole FA94-9 drilled through the mafic-ultramafic assemblage and into the sediments with no indication of an alteration zone or quartz vein system.

Holes FA94-6 and FA94-7 were located 50 metres on either side of hole FA94-5 (Fig. 6.3). Hole FA94-6 intersected gold mineralization in ultramafic rock and a section assayed 5.94 g/t Au over 0.5 metre. A section of sericite, carbonate, silica altered mafic rock assayed 3.74 g/t Au over 1.5 metre in hole FA94-7.

Two geophysical programs were completed during the 1994 exploration program. Both geophysical programs included ground magnetics and a three-frequency Horizontal Loop Electromagnetic (HLEM) survey.

After completing the drilling program, 192 new claims were staked in May 1994 to the north, south and west of the Fenelon "A" property. In addition, other claims blocks in the vicinity (Gaudet "C" and Gaudet "A") were annexed to the Fenelon "A" property. At this time, the Fenelon "A" property was represented by 448 staked claims. On April 30, 1994, a new Joint Venture agreement (Fenelon "A" Joint Venture: "FAJV") was signed between Cyprus and OGY, thereby replacing the CBJV.

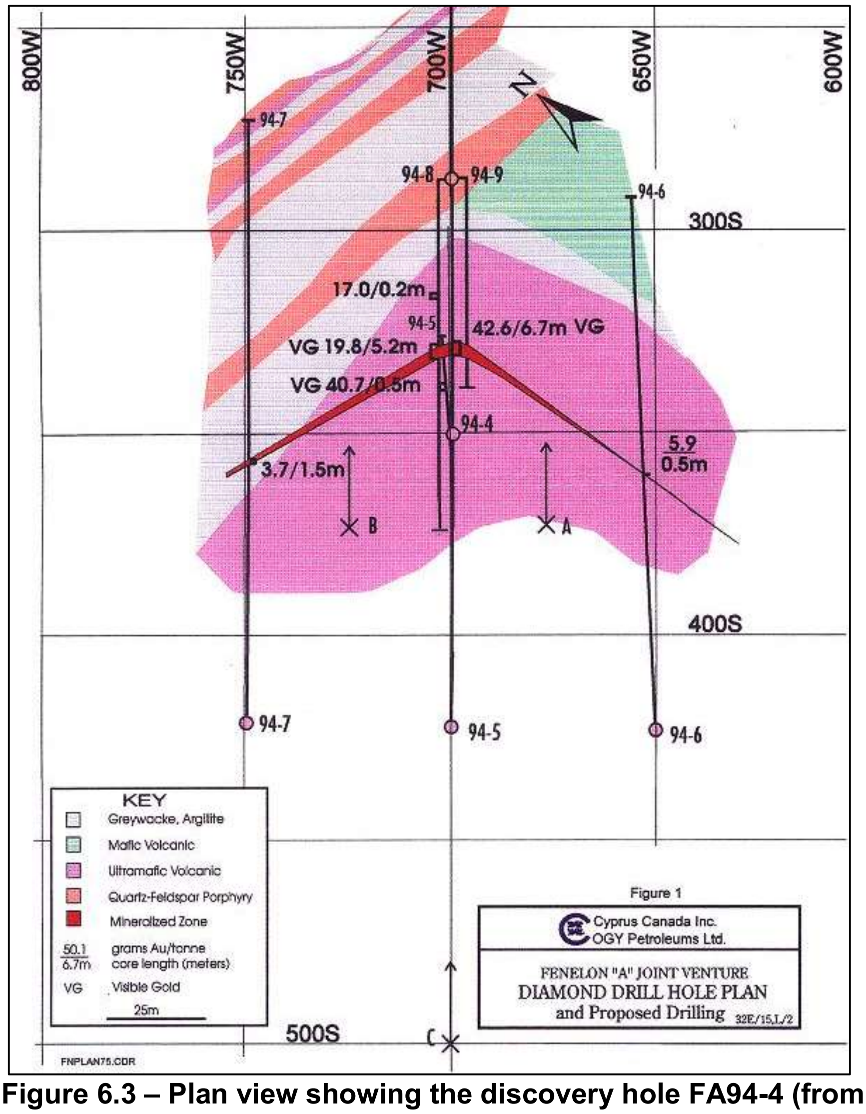


Figure 6.3 – Plan view showing the discovery hole FA94-4 (from Guy, 1994)

6.5 1995 Exploration Program (Cyprus-OGY)

The winter 1995 exploration program on the Fenelon “A” property included diamond drilling, limited claim staking, drill hole surveying (both surface and downhole) and an orientation IP survey (Needham and Nemcsok, 1995).

The diamond drill program, including fifty-seven (57) BQ-size drill holes (FA95-10 to FA95-65) totalling 13,374 metres, was performed from December, 1994 to April, 1995. Several significant gold intersections were obtained near surface over a strike length and depth of approximately 250 metres. Visible gold has now been observed in 18 drill holes. Some of the better intersections include: 14.24 g/t Au over 13.9 metres, 9.78 g/t Au over 7.2 metres, 13.74 g/t Au over 6.8 metres and 37.48 g/t Au over 6.99 metres. Gold mineralization shows good correlation with chalcopyrite mineralization and also copper ICP analyses. Some correlation was observed for arsenopyrite mineralization but not necessarily as ICP analyses. Pyrrhotite mineralization has the tendency to be stronger within the gold mineralized zone.

The best gold intersections are associated with strongly silicified, sometimes “cherty” appearing alteration zones that cut across stratigraphy. The strike and plunge extensions of these significant intersections are interpreted to be displaced by N-NNE-trending block and/or thrust faulting. Faulting has made the interpretation of the plunge of the zone difficult to define. In addition, the presence of multiple silicified horizons on each section made the interpretation of the “zone” difficult unless visible gold was actually observed in the core. Downdip/plunge continuity problems were encountered on some of the cross sections. As interpreted at that time, the zone has a variable strike ranging from 105° to 140° dipping to the southwest at approximately 80°. The Zone is thought to be associated with a brittle break, not a ductile shear zone, and may be spatially associated with a southwest-dipping quartz eye porphyry unit. In addition to the Main Zone, a footwall zone (FW) and three hanging wall zones (i.e. HW1, HW2 and HW3) were intersected. The gold mineralization associated with these zones did not appear to be as broad or strong as that intersected in the Main Zone.

Sperry Sun’s single-shot azimuth tests proved to be unreliable, apparently due to bedrock magnetics as discovered after performing a 23-hole gyroscopic survey following the completion of the winter drill program.

An orientation IP survey was completed over the Discovery Zone for a total of 3.5 kilometers (Lortie, 1995). The Discovery Zone is associated with a “shoot” running off a strong resistivity high adjacent to a strong chargeability anomaly, and correlates with a moderate magnetic low break in both the ground and airborne magnetic surveys.

6.6 1995–1996 Exploration Program (Cyprus-Fairstar)

Effective July 1, 1995, OGY made an agreement with Fairstar Explorations Inc. (“Fairstar”) transferring all of OGY’s interests in the CBJV to Fairstar, including the FAJV (Fig. 6.4). Cyprus is always the operator of the FAJV.

From October 1995 to January of 1996, exploration program included 241.7 line kilometres of line cutting and geophysical surveys (Needham and Nemcsok, 1996). The purpose of this program was to define new targets, similar to the Discovery Zone. The work included 183 kilometres of frequency domain IP surveys, 31 kilometres of HLEM and 241.7 kilometres of combined magnetic and VLF surveys (Boileau and Lapointe, 1996).

The 1995–1996 Fenelon "A" diamond drill program consisted of thirty-six (36) diamond drill holes (FA95-66 to FA95-77 and FA96-78 to FA96-101) and the extension of two previous diamond drill holes totalling 9,851.47 metres (Needham and Nemcsok, 1996). Of this footage, a total of 6,454.5 metres in twenty-three (23) diamond drill holes was completed on the Discovery Zone. A total of 3,397 metres in fifteen (15) diamond drill holes was completed as "Wildcat" reconnaissance diamond drill holes. A total of thirty-one (31) holes of an attempted thirty-four (34) were surveyed downhole using the gyroscopic method (surveys by Sperry Sun and CBC Wellnav). In addition, Descarreau and Dubé completed collar azimuth surveys on forty-eight (48) of the diamond drill holes in the Discovery Zone area.

Two holes (FA97-102 and FA-97-103) totalling 540.4 metres were drilled outside the Discovery Zone area.

The auriferous portion of the main zone appeared to be cut off. The potential contained ounces in the Main Zone, did not meet Cyprus' minimum requirements.

6.7 1996–1997 Exploration Program (Fairstar)

In October 1996, Fairstar became operator of the FAJV and incurred exploration expenditures on the order of C$2 million over the course of the 1996–1997 winter field program on the Property (Kelly et al., 1997). Cyprus did not contribute to this exploration program and as a result, the Fairstar and Cyprus interests became approximately 70% and 30% respectively.

Between January 6, 1997 and April 7, 1997, seventy-seven (77) holes (FA-97-102 to FA-97-178) were drilled on the Fenelon "A" property for a total of 15,924.4 metres.

The field activities of the program were conducted between October 1996 and April 1997. On the Discovery Zone, thirty-eight (38) diamond drill holes were bored during the program for a total of 6,497.8 metres (Kelly et al., 1997). The objectives of this drilling were to define the limits of the Discovery Zone, provide for 25-metre hole spacings within the zone and to improve understanding of the geometry of the mineralization and of the nugget effect. A re-interpretation of the Discovery Zone, based on the extensive Foster core orientation tests, showed the mineralization to be made up of eight (8) east-west "en echelon" gold-bearing structures associated with an ultramafic intrusion having an overall northwest orientation. The new model of the Discovery Zone greatly enhanced the understanding of its structure and geology, and it was thought at the time it would facilitate the future task of extending the zone to depth and along strike. The mineralized zones had thus far been investigated in detail over 275 metres in length and to a depth of some 200 metres.

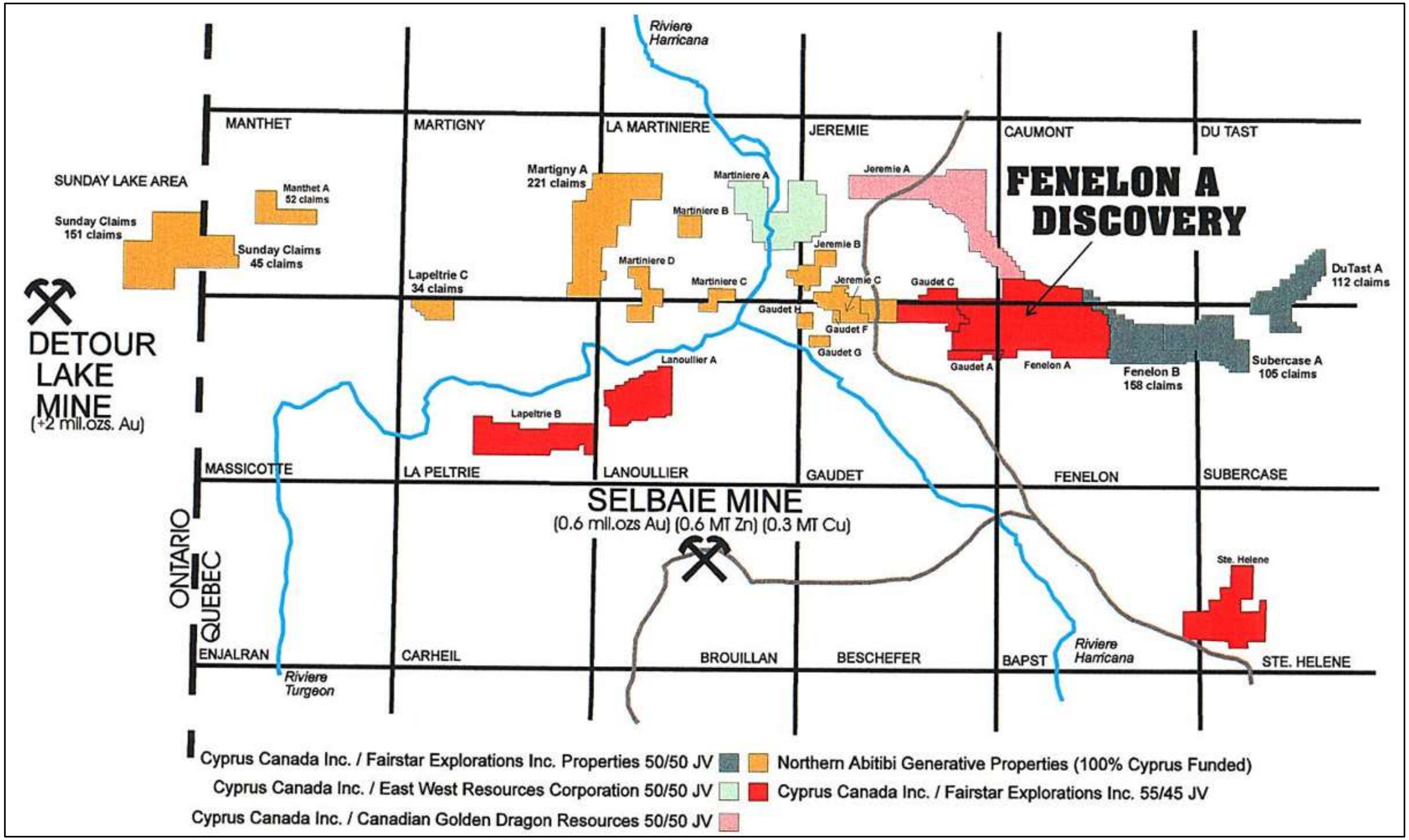


Figure 6.4 – Location of Fairstar's properties after the transfer of OGY's interest (from Needham and Nemcsok, 1996)

In addition to diamond drilling, a geotechnical investigation was carried out to test the thickness and nature of the shallow overburden covering the Discovery Zone. This work included a detailed seismic refraction survey (Poulin and Goupil, 1996) and five (5) holes drilled to specifically test the physical characteristics of the overburden.

Exploration elsewhere on the FAJV demonstrated the potential of other areas. In 1997, line cutting (92.7 km), Mag (72.7 km) and IP (107.2 km) surveys were carried out (Boileau, 1997), and thirty-nine (39) diamond drill holes were drilled for a total of 9,426.6 metres.

In November 1997, Fairstar announced they had received a positive pre-feasibility ("PFS") report on the Discovery Zone of the FAJV (Fairstar press release of November 13, 1997). The study, prepared by CHIM International ("CHIM"), a Montreal based geological consulting firm, was designed to confirm the resources, establish an appropriate grade cutting procedure in light of the relatively strong nugget effect pervasive throughout the deposit, develop a conceptual plan to exploit the deposit and establish the financial viability of the project.

CHIM audited the resource estimation done by Géospex Sciences Inc. and updated them to "reserves". A new estimate by polygonal method was prepared incorporating a minimum mining width of 2 metres and capping high grades to 100 g/t Au on individual assays. The revised estimate prepared by CHIM indicated a resource (uncategorized) of 252,000 tonnes averaging 14.2 g/t Au for a total gold content of 115,000 ounces. The zone has an average thickness of 2.68 metres.

These "resources" are historical in nature and should not be relied upon. It is unlikely they comply with current NI 43-101 criteria or CIM Standards and Definitions, and they have not been verified to determine their relevance or reliability. They are included in this section for illustrative purposes only and should not be disclosed out of context. InnovExplo did not review the database, key assumptions, parameters or methods used by CHIM for this mineral resource estimation on the Discovery Zone.

Preliminary metallurgical tests were carried out at the *Centre de Recherche Minérale* in the city of Québec. The tests were based on a 20 kilogram representative sample derived by quartering the existing core. These tests show the processing of the gold-bearing material to be straight forward with no harmful elements arising from the treatment process. The gold recovery ranged from 96.5% to 99.1%, depending on the type of metallurgical test used. The work index has been calculated at 10.5 kWh/t, another very favourable characteristic.

The conclusion reached in the CHIM report was that the project, at current gold prices, was economically viable. Assuming the price of gold at US$320/oz and taking into account refining charges and royalties, the operating cost was calculated at US$187/oz. The financial analysis indicated a cash flow of C$8.0 million, a rate of return of 67% on a pre-tax basis and an NPV of $5.0 million using a 12% discount rate. The payback period was 17 months after the start of production.

This "PFS" is historical in nature and should not be relied upon. In 1997, it was considered NI 43-101 compliant. Since 1997, more drilling has been added and more geological information has become available. Additionally, assumptions used to determine cut-off grades as well as estimated capital and operating costs are likely to have changed since 1997. Consequently, this "PFS" cannot be considered as current.

It is included in this section for illustrative purposes only and should not be disclosed out of context.

The report's recommendation was to begin the necessary permitting work (including the execution of a preliminary impact study) to conduct a bulk sampling program in order to confirm the grades and recoveries, with the ultimate goal of mining the deposit by way of open pit. The pit would be 70 metres deep and the total amount of ore to be mined would be 137,000 tonnes at an average grade of 17.5 g/t Au, netting 77,000 ounces. The mining rate would be 4,000 tpd, resulting in a mine life of approximately 3 years. The waste/ore ratio would be 15.6/1. Little infrastructure and capital costs would be required as all installations would be temporary and provided by contractors. Electrical power would be sourced by on-site generators.

6.8 1998–2000 Exploration Program (Taurus-Fairstar)

In July, 1998, International Taurus Resources Inc. ("Taurus") announced the signing of a formal agreement with Cyprus whereby Taurus acquired a 100% interest in Cyprus' share of a portfolio of twenty (20) properties in the Casa Berardi sector, including the Fenelon "A" property or FAJV. At this time, Taurus controlled approximately 30% of the Fenelon "A" property (Fig. 6.5) through the Cyprus agreement.

During 1998, Fairstar developed the access road to the Discovery Zone site in preparation for a proposed bulk sampling program. Fairstar also completed a drill program in 1998, testing the up-dip projection of the zone to the bedrock- overburden interface (Guy and Tims, 2000). The objective of this program was to prepare for a stripping and bulk sampling program in order to evaluate the continuity of the gold zone in preparation for mining of the high-grade zone. Holes for this program were not in the sequential order for 1998, but were recorded after the year 2000 hole numbers, as the results of this program were not known at the time the 2000 program was conducted. The 1998 holes were not marked in the field and the JV partners (Taurus-Fairstar) were not apprised of the program. The 1998 Fenelon "A" diamond drilling program consisted of six (6) short holes (FA-98-178 to FA-98-182A, FA-98-182B and FA-98-183) totalling 200.9 metres.

In May 2000, Fairstar granted to Taurus an option to increase its interest in the FAJV by financing certain exploration expenditures, including the collection and processing of a bulk sample.

Taurus became operator of the FAJV. The 2000 exploration diamond drilling program conducted by Taurus ran from September 9 to October 12, 2000. The program consisted of twenty-four (24) NQ-size drill holes (FA-00-179 to FA-00-201, including FA-00-196A) totalling 992.4 metres (Guy and Tims, 2000). The holes were drilled on the Discovery Zone where previous work by Cyprus and Fairstar had outlined a resource of 252,000 tonnes at 14.2 g/t Au for a total of 115,000 ounces of gold (see Fairstar press release of November 13, 1997).

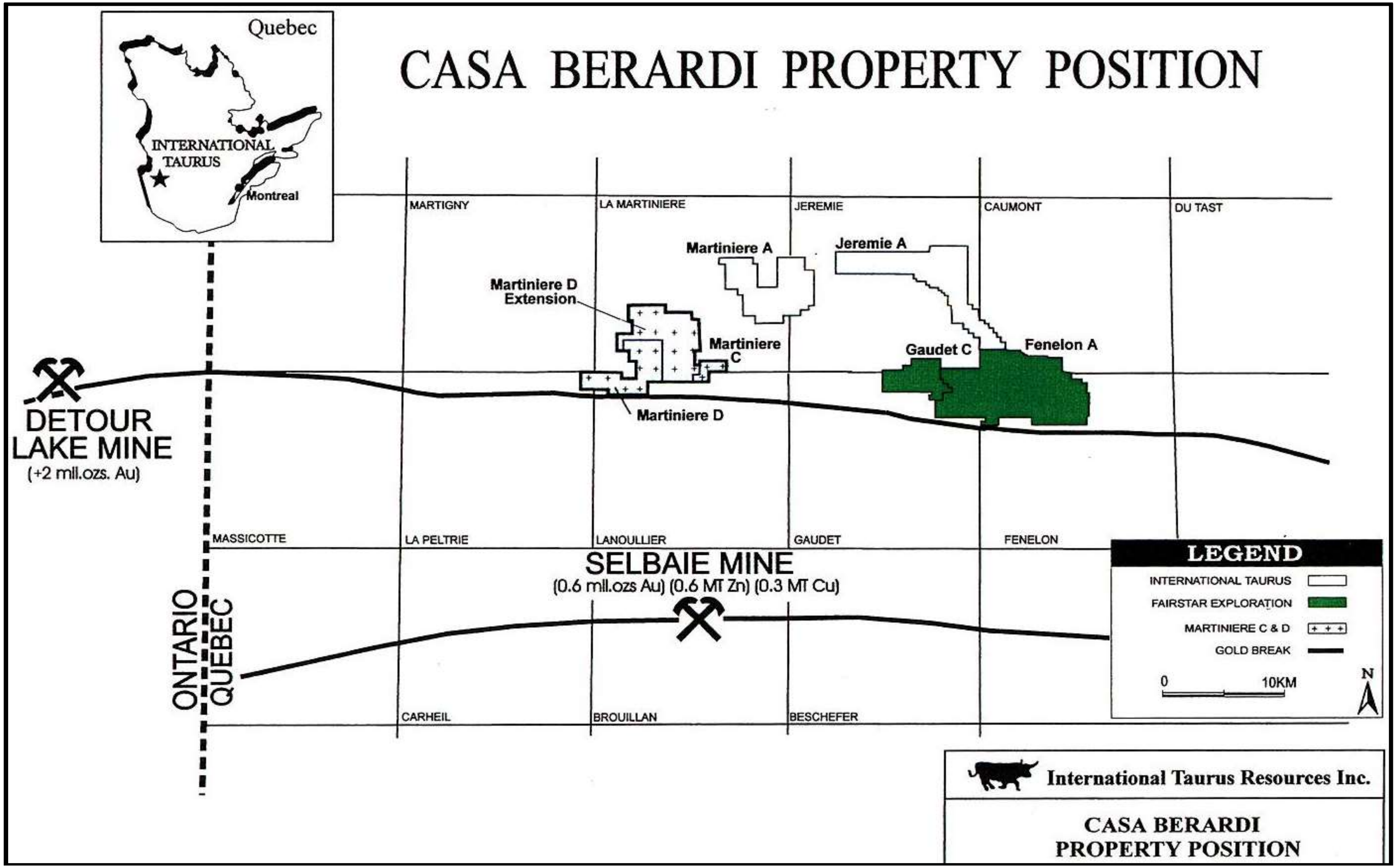


Figure 6.5 – Location of Taurus's properties after the transfer of Cyprus's interest (from Guy, 2001)

The objective of the 2000 drill program was to trace the known gold mineralization to the bedrock–overburden interface to plan a stripping and bulk sampling program for the mineralized zone. This would establish the confidence in the continuity of the mineralization necessary to undertake the mining of the resource. Drilling was concentrated on the 2S zone using seventeen (17) holes, where the majority of the resource has previously been delineated. Results indicated very erratic mineralization in the vicinity of previous intersections. The mineralization was not in a planar dipping sheet geometry, as indicated by the lack of ability to follow intercepts in any direction. Drilling on the 0S, 3S and 4S veins failed to locate quartz veins in close proximity to previous wide intercepts with visible gold. These veins were poorly defined by the previous work and the closely spaced testing of the 2000 program indicated the interpreted attitude was possibly incorrect. Drilling on all the veins indicated a lack of continuity as interpreted during the previous exploration work. Drilling on vein structures between holes failed to intersect the vein as predicted in the proposed model. The shallow overburden cover, erratic nature and extremely high grade of the veins at the Discovery Zone indicated that the most definitive and cost effective method to further explore the zone was by stripping.

6.9 2001 Exploration Program (Taurus-Fairstar)

A bulk sampling program was conducted by Taurus on the Discovery Zone from February to June, 2001 (Veilleux, 2001; Guy, 2001). The contract for overburden removal and all related work was awarded to Fournier & Fils of Val-d'Or, and Castonguay & Frères (Forage Nord-Ouest Inc.) was awarded the contract for drilling and blasting. The ore was loaded on trucks for transport to the Camflo Mill, owned by Richmont Mines Inc.

The objective of the 2001 bulk sample program (Fig. 6.6) was to test-mine the 0S, 1S and 2S gold zones to obtain information that would assist in the preparation of a feasibility study. The program would also establish the necessary confidence in the continuity of the mineralization to undertake the mining of the resource. The overburden was stripped and the outcrop surface was mapped and sampled. Mining concentrated on the 1S and 2S zones where the majority of previous work had been conducted and the larger resource had been delineated.

Once the surface area was washed and stripped, the 1S and 2S mineralized zones were mapped and sampled. A total of seventy-four (74) channel samples were collected, ranging in length from 0.4 to 2.1 metres.

An intermediate zone between 1S and 2S and east of 2S, named the VI zone, was also mapped and sampled. The 0S zone was not significantly mineralized at surface, but high-grade mineralization was located in the northwest wall of the open pit and mined as part of the bulk sample exercise.

Two types of mineralization are noted:

- Interflow volcanic sediment-hosted, typified by the 1S zone, with mineralization grading as high as 187.96 g/t Au and averaging 111 g/t Au (from samples taken from mineralized muck); and

- Shear-related mineralization, typified by the 0S, VI and 2S zones, with higher gold values of up to 926.75 g/t Au, averaging 537 g/t Au (from samples taken from mineralized muck).

Both types of mineralization are related to the volcanic contacts where an inherent zone of weakness and increased porosity has served as a fluid conduit and a location for shearing.

The **0S mineralized zone** was observed and mapped on surface as a carbonatized, chlorite-rich volcanic interflow unit. It was not significantly mineralized anywhere along the surface expression, however mineralization occurred a few metres below surface where the interflow aspect has mostly pinched out to a sheared volcanic contact with shearing becoming more intense with depth. The zone represents shear-related mineralization. High-grade quartz-pyrrhotite-chalcopyrite mineralization (81.98 g/t Au over 1.0 m) was located 3 metres below the surface expression in the northwest wall of the pit. This zone was then mined over a length of 16 metres, a depth of 3.5 metres and a width of 2.5 to 3 metres. Very little mineralization from the 0S zone remains in the open pit. The zone was only mined to the 1st level and no mineralization was noted in the floor at that elevation. Limited previous drilling in the vicinity of the 0S zone failed to locate the zone beyond the two holes used to define the zone, and very little drilling has been conducted to explore for this zone along strike or downdip. The amount of high-grade mined ore exceeded expectations based upon the closely spaced drilling in the mined out area. With very few drill intercepts along strike, it was felt that the 0S zone may have potential for more mineralized pods along the horizon, similar to the other mined zones, which contained multiple mineralized pods both along strike and downdip.

The **1S mineralized zone** was mapped on surface as mineralized, carbonatized and chert-rich interflow volcanic sediment. The interflow unit hosts sheared and silicified en echelon pods of pyrrhotite, chalcopyrite and gold mineralization. The 1S zone was mined in all three levels of the pit over a strike length of 37 metres, a maximum width of 5 metres and a height of 16.5 metres. Small amounts of 1S ore remain in the pit area: in the floor of the 3rd level, in the pit wall to the north and east, and as pods extending to the west, where the zone is exposed in the bench excavated for the 0S zone. The 1S zone remains open at depth below the pit floor as indicated in earlier drill holes and verified by exposures in the mined lower level. Previous drilling indicates the 1S zone continues to the east beyond the east wall of the pit and to the west. The linear continuity of the interflow structure in three dimensions, as observed during mining operations, suggests that the 1S zone should persist and present a recognizable target for drilling. The high-grade nature of the mineralization and the close spacing of the pods along the zone suggest that the zone can be mined as a continuous body allowing for internal dilution.

The **VI or Intermediate mineralized zone** was mapped on the surface as a sheared, carbonatized interflow unit with mappable sections of silicification, pyrrhotite and chalcopyrite containing very high concentrations of gold. The VI zone was mined on all three levels of the pit. The zone was mined in conjunction with the 2S zone and on the 3rd level with the 2S and 1S zones, resulting in excessive dilution. The VI zone remains open to the east of the pit and at depth. High-grade mineralization remains in the southeast wall and the east wall of the pit. At the surface, the wider and higher-grade mineralization was traced to the edge of the pit where it continued to the east under the overburden. Limited drilling indicates that the zone persists to depth. The west end of the VI zone overlaps and merges with the 2S zone where it was interpreted to be the 2S zone, resulting in 2S drill intercepts that were exceptionally wide.

The **2S mineralized zone** was mapped on the surface as a silicified, carbonatized, pyrrhotite- and chalcopyrite-rich shear zone within both a quartz-feldspar porphyry (QFP) body and the dominant host mafic volcanic rocks. The 2S mineralized pods are of a greater volume than those of the other zones mined in the bulk sampling program. Most of the mineralization mined from the 2S zone was from one pod, which measured 17 metres long by 6 metres wide by more than16 metres high. However, the zone was not pervasively and homogenously mineralized, particularly in the volcanic rocks, where unmineralized pillows and/or volcanic blocks constituted large waste blocks within the ore. As previously, the mineralization and shearing followed the flow contacts, with the more pervasive alteration and mineralization occurring along these planes of permeability and weakness. Blocks of unmineralized material within the ore horizon (i.e., internal dilution), were visually estimated over approximately 50% of the structure. The 2S zone remains open at depth with existing drill holes intersecting the zone at least 35 metres below the present pit-floor elevation. The zone also remains open to the south, below and around the QFP, as indicated in the pit walls, and to the east and west, including both the QFP contact and the continuation of the shear in the volcanics.

The 0S, 1S, VI and 2S zones were all larger and more continuous than postulated from the drill data. This was due to the short strike length of the high-grade pods within the zones and the fact that many of the drill holes apparently intersected pinched-out areas or internal waste blocks within the mineralized structures. This resulted in the mineralized zones appearing to be extremely erratic. The geometry of the zones also made it difficult to interpret, using drill holes angled into the structure, due to the gash-vein and pod-like nature of the high-grade mineralization.

Although the ore on the Discovery Zone is extremely high grade, the mill results were considerably lower due to excessive dilution, which was caused by the mining method used by the bulk sampling program. The open pit mining method could work with this type of mineralization; however, not as a bulk mining scenario. The 5.5 metre bench height used exceeded the height of many of the en echelon pods. The minimum width of the blasts was an arbitrary 3 metres in ore and 5 metres in waste, which generally exceeded the width of the mineralization in the zones. Most of the blast lengths also exceeded the strike length of the high-grade pods. No attempt was made to slash the waste to the ore contact, nor was the ore efficiently slashed prior to the waste round. Because the location of mineralized pods was not well known, these techniques resulted in more than 100% external dilution. Due to the proximity of the zones to each other, the 2nd and 3rd levels were taken in their entirety in an attempt to “bulk mine” the entire pit. This resulted in an internal dilution estimated to be from 100% to 200%, and external dilution in the order of 500%.

A mining summary from the mining operation on the Discovery Zone is provided in the report of Veilleux (2001), from which the following description is taken.

A total of 107,000 m^3 of overburden were removed and stockpiled in a designated area. For disposal of the overburden, an area of about 370 metres by 115 metres was cleared of all trees. The thickness of removed overburden ranged from 5 to 11 metres. A total of 71,680 tonnes of blasted rock (waste and ore) was extruded from the open pit. From this total, 11,603 tonnes of waste were necessary to construct the retaining dyke. The total of ore before sorting corresponded to 18,966 tonnes. A total of 5,131 tonnes was discarded from blasted ore zones.

The total of ore loaded and shipped to the Camflo mill represents 13,835.3 wet metric tons or 13,752.3 dry metric tons. After milling, a total of 4,245.21 ounces (132,038.77 g) was produced at a recovery grade of 9.60 g/t Au, corresponding to a recovery of 97.03% (Veilleux, 2001).

Figure 6.6 – Bulk sample program conducted by Taurus on the Discovery Zone (photo from Balmoral's website)

A crude and rough visual sorting of the ore and waste took place in the pit prior to loading from the pit as well as on the oversize muck on the ore piles prior to breaking and trucking. It is not possible to estimate what portion of the 5,131 tonnes that was sorted out was internal dilution as opposed to external dilution. It must be stated that the muck pile sorting was only conducted on the oversize material in the muck pile. This constituted a very small proportion of the total muck, probably in the order of 10–15% (Guy, 2001). Of the oversize material, it was estimated that only 25% was ore and 75% was waste. Using that formula, it suggests that the total tonnage of mineralized rock or "ore" was 4,500 tonnes. That number would exclude both internal and external dilution.

According to Guy (2001), in a more efficient mining scenario, it would be possible to mine the mineralized zones or structures including internal dilution and the easily visible contacts of the zone would allow for minimal external dilution. Available data based on the geological mapping indicates that the tonnage of the mineralized zones, including internal dilution and a minimum mining width of 1.5 metres, could possibly have been in the order of 8,700 tonnes as opposed to the 18,966 mined for ore.

On October 16, 2001, Taurus acquired a 66.67% interest in the FAJV and Fairstar retained a 33.33% interest.

Pincock, Allen and Holt Ltd. (“PAH”), a division of Hart Crowser Inc., was retained by Taurus on behalf of the FAJV in October 2001 to prepare a new resource estimation and scoping study on the Fenelon Gold Project, evaluate the pilot-mining project proposed by Taurus, and provide recommendations for additional work to advance the FAJV to the feasibility stage (Poos et al., 2002). PAH did not visit the Property or examine any core from the Property. The scope of work did not include reviewing the environmental regulations relative to the pilot-mining project or the metallurgical characteristics of the Property. Discussions with Taurus project personnel were held in Vancouver and Denver.

A grade model was developed by PAH that would recreate the results obtained from the previous bulk-sampling program. Generation of the model was based on geologic interpretation. Two different sets of interpolation parameters were used in order to represent the two different structural orientations of mineralization. This model was within 1% of the results of the bulk-sampling program. The remaining indicated resource in the composite capped model was 168,000 tonnes at a grade of 5.29 g/t Au for a total of 28,600 contained ounces. PAH estimated the initial, base case, pilot-mine in-pit indicated resource as 44,000 tonnes grading 6.74 g/t Au for a total of 9,500 contained ounces.

These “resources” are historical in nature and should not be relied upon. It is unlikely they comply with current NI 43-101 criteria or CIM Standards and Definitions, and they have not been verified to determine their relevance or reliability. They are included in this section for illustrative purposes only and should not be disclosed out of context. InnovExplo did not review the database, key assumptions, parameters or methods used by PAH for this mineral resource estimation on the Discovery Zone.

PAH designed a pit for the initial pilot-mining program based on pit slopes of 6H:1V in the humus and till, and 55° overall in the bedrock. The unit costs from the bulk-sample pit were used by PAH as a starting point for the operating cost estimates. These costs were decreased based on the assumption that because of the larger tonnage being excavated and processed, a lower unit price could be negotiated. The analysis of the cash flows indicated that by reducing the dilution and improving the grade control, the initial pilot-mining project had the potential to generate revenue in excess of costs of up to C$800,000.

This “Scoping Study” is historical in nature and should not be relied upon. Since 2002, more drilling has been added and more geological information has become available. Additionally, assumptions used to determine cut-off grades as well as estimated capital and operating costs are likely to have changed since 2002. Consequently, this “Scoping Study” cannot be considered as current. It is included in this section for illustrative purposes only and should not be disclosed out of context.

6.10 2002–2004 Exploration Program (Taurus-Fairstar)

In September 2002, the pilot-mine excavation started on the Discovery Zone from the Fenelon "A" Property. The contract was awarded to Construction Norascon Inc. of Amos, Québec. Stripping of overburden silt and till exposed an area of bedrock measuring 70 by 180 metres (Fig. 6.7). The bedrock was washed, mapped and sampled (channel sampling) in detail to determine the distribution and controls of the mineralization. This work was conducted by Christian Derosier, P.Geo., of SRK Consulting ("SRK"), international geologists and consultants. A structural analysis was conducted on the stripped area. The new stripped area and the 2001 open pit (bulk sample) were also surveyed. A total of nine hundred sixty-three (963) channel samples were collected, varying in length between 0.2 and 2.1 metres.

Figure 6.7 – Stripping work on the Discovery Zone (from Derosier, 2003)

From October 20 to November 22, 2002, a diamond drilling program was undertaken on the Discovery Zone. A total of forty-two (42) short holes (FA-02-207 to FA-02-248) of NQ diameter core totalling 2,351.0 metres were drilled. Drill holes were bored from the surface rock or from the bench built around the stripped area. All collars were surveyed by the Norascon's surveyor. Diamond drill holes were targeted to intersect the known mineralized zones at a depth not exceeding -50 metres vertical. The aim was to better control the location and size of the mineralized zones at depth, as well as their plunge. Results of this program were expected to lead to a calculation of mineable resources on the southwest extension of the open pit.

SRK was retained by Taurus and Fairstar to generate a geological model and a new mineral resource estimate on the Discovery Zone. A 43-101 compliant technical report was prepared (Couture and Michaud, 2003).

SRK reviewed, repaired and updated the database, consisting primarily of 195 drill holes and extensive surface channel sampling. Given the QA/QC programs employed over the various exploration campaigns, SRK was confident in the reliability of the data. According to SRK, the key factors affecting estimation of the mineral resources for the Discovery Zone are the interpreted variable geometry of the higher-grade portions of the deposit and the presence of high-grade gold values, often exceeding 100 g/t Au. SRK’s geological model describes a central zone of mafic rocks flanked by argillaceous sedimentary units. Within this central zone, strong alteration, including silica and sericite with carbonate, is associated with variable amounts of sulfide and quartz veining with gold in several mineralized zones. These zones, as indicated by 195 drill holes, are over 100 metres in combined width, extend at least 200 metres along strike and to at least 300 metres in depth. The area in which SRK measured the bulk of its estimated resource occurs along a strike length of 110 metres in the upper 50 metres of the deposit. It includes four of the nine originally reported major gold-bearing vein-like structures. This is the area of greatest drill-hole density and it represents a small portion of the Discovery Zone area.

The modelled gold mineralization occurs within the mafic unit and along its contact with the argillaceous sediments. SRK has adopted an interpretation in which the bulk of the mineralization of the core area is contained within six separate zones of alteration and gold-sulfide mineralization. In SRK's view, the bulk of the high-grade gold intercepts reported during earlier programs occur as irregular zones within broader alteration halos. Using ordinary kriging, grade capping (2m composites capped at a maximum of 50 g/t Au within the central HW Zone) and Gemcom® programs, SRK constructed and interpolated gold grades into a 3D model. This model extends across the broader zones of alteration, or domains, which can be confidently constructed from the available data. SRK did not join drill holes, which contained zones of higher-grade gold mineralization, based solely on assay data. SRK used this information to construct three-dimensional solid bodies to represent the strike and down-dip extensions of the alteration zones and their attendant high- and low-grade gold mineralization.

The SRK resource differs from that of previous estimators, whose interpretation of the mineralized zones assumed greater continuity between the higher-grade portions of the alteration zones that define narrower and more tabular zones. SRK also built three other models using different interpolation methods: ordinary kriging uncapped, indicator kriging uncapped and ID3 capped. In SRK’s opinion, the ordinary kriged and capped model best represents the mineral resource (Table 6.1).

Table 6.1 – 2003 SRK Mineral Resource at a cut-off grade of 5 g/t Au (from Couture and Michaud, 2003)

	Tonnes	Grade	Contained Gold
	(x1000)	(g/t Au)	(oz)
Indicated	49.55	11.24	17,900
Inferred	38.84	10.49	13,100

These "resources" are historical in nature and should not be relied upon. In 2003, they were compliant with applicable NI 43-101 criteria and CIM Standards and Definitions. Since 2003, more drilling has been added and more geological information has become available. Additionally, assumptions used to determine cut-off grades are likely to have changed since 2003. Consequently, these "resources" cannot be considered as current. They are included in this section for illustrative purposes only and should not be disclosed out of context.

In April 2003, Taurus owned a 62% interest in the project and Fairstar retained a 38% interest.

Mineral Resources Engineering of Murray, Utah, was contracted in June 2003 to design and cost an underground development project to be part of a Preliminary Assessment Study ("PAS") and a mining test of high-grade gold mineralization at the Discovery Zone on the Fenelon "A" Property (Drips and Bryce, 2003; 2004). This study included the detailed design of a ramp and associated infrastructure to provide access to the mineralized bodies identified by SRK Consulting in their study dated April 2003. Mineral Resources Engineering evaluated the potentially extractable gold resources generated using a polygonal estimation method, rather than computer modelling (kriging). Mineral Resources Engineering did not classify the resources. The resource estimate does not comply strictly with the requirements of NI 43-101, but was used to generate possible scenarios for internal planning and budgeting.

The project schedule had three phases, which started in the third quarter of 2003 and terminated in the fourth quarter of 2005. The base case mining rate was 250 tpd. The total cost for the base case project, as defined, was C$12,214,309 and the anticipated return from processing the 92,147 tonnes was C$13,698,246 (based on the assigned grade of the resource, the dilution, and a gold price of C$480/oz). The base case project, as defined in this study, generated an IRR of 43.7%, or a NPV of C$813,505.

This PAS is historical in nature and should not be relied upon. Since 2003, more drilling has been added and more geological information has become available. Additionally, assumptions used to determine cut-off grades as well as estimated capital and operating costs are likely to have changed since 2003. Consequently, this PAS cannot be considered as current. It is included in this section for illustrative purposes only and should not be disclosed out of context.

This design of the PAS was used as the basis for solicitation of bids for mine construction from qualified area contractors. Following a round of competitive bidding for the construction of the underground access and test mining, a contract was awarded to Ross Finlay 2000 Inc. of Val-d'Or. Work on the underground project commenced in mid-October 2003 and a full camp and support facility were established.

The underground exploration program undertaken in 2003 and 2004 consisted of driving of a decline from the stripped outcrop to the zones interpreted from surface work (Pelletier and Gagnon, 2004). The portal of the ramp started in the north wall of the open pit (Fig. 6.8) and a decline was driven down at 15% grade over 326 metres. It provided the access needed to develop more than 745 metres of drifts, crosscuts and raises, of which 254 metres were driven in ore.

This development in the ore generated a volume of 5,374 t at 16 g/t Au (mostly the muck from sills and breasts) over widths of at least 1.5 metre. Lower grade material was also recovered (800 t at 3.0 g/t Au) in crosscuts averaging 4.5 metres wide. All development material was stockpiled on surface to be processed in the near future. Those developments generated 359 face samples, 258 test hole samples and 149 muck samples. Those developments also generated sufficient 3D information to confirm the shape of the lenses of mineralized material, the lateral maximum extent and the continuity. New information on structural and lithological controls was also obtained (such as a shear zone cutting the “C” mineralized zone). Definition diamond drilling was also performed during this underground exploration program. A total of fifty-four (54) underground NQ-size holes were drilled from the northern access drift on level 5213 for 3,975.5 metres. The holes were drilled on a spacing of 5 to 10 metres. Eight additional holes (BZ-04-001 to BZ-04-029; 78m) were also drilled from production drifts.

Figure 6.8 – Portal of the ramp started in the north wall of the open pit (from Balmoral’s website)

During 2004, InnovExplo completed an updated resource estimate on the central Discovery Zone corresponding to an area approximately 160 metres long, down to a depth of 175 metres (Pelletier and Gagnon, 2004). Using data from current and previous drilling, as well as the ongoing mapping, sampling and other work by site geologists, InnovExplo was able to confirm that the sampling methodology, assaying methods, database management and core logging were carried out according to standard industry practices.

Using a 5.0 g/t Au minimum cutoff, the contained ounces in the combined measured and indicated categories amounted to 35,107 ounces of gold at an average grade of 19.61 g/t Au, with a further 11,204 ounces at an average grade of 12.79 g/t Au in the inferred category. Of the total measured and indicated resource, 4,002 tonnes grading 18.36 g/t Au was classified as measured and 52,255 tonnes grading 19.71 g/t Au as indicated.

In estimating the resource to be used for outlining potentially minable blocks, InnovExplo used a polygonal method in the plane of the veins. Following a rigorous statistical evaluation of the database and an adoption of a conservative stance for the evaluation, high values were capped at 50 g/t Au for blocks determined to be in the measured category, and at 75 g/t Au for blocks in the indicated and inferred categories. Drill-hole intersections were diluted out to a minimum horizontal width of 1.5 metres, but no further edge dilution was applied.

Table 6.2 – 2004 InnovExplo Mineral Resource at a cut-off grade of 5 g/t Au (Pelletier and Gagnon, 2004)

Uncapped							Capped					
Cutoff grade, g/t	Metric tonnes			Metric tonnes			Metric tonnes			Metric tonnes		
		Measured + Indicated			Inferred			Measured + Indicated			Inferred	
		Grade, g/t gold	Ounces gold		Grade, g/t	Ounces gold		Grade, g/t gold	Ounces gold		Grade, g/t	Ounces gold
0	**309,131**	**7.38**	**73,348**	226,229	2.46	17,895	**309,131**	**3.83**	**38,066**	226,229	1.86	13,350
3	**65,032**	**34.23**	**71,569**	28,918	11.45	10,647	**65,026**	**17.32**	**36,210**	28,918	9.85	9,159
5	**56,257**	**38.99**	**70,521**	27,388	17.68	15,570	**55,684**	**19.61**	**35,107**	27,245	12.79	11,204
7	**49,929**	**43.21**	**69,363**	19,886	22.28	14,232	**49,356**	**21.40**	**33,958**	19,743	15.47	9,821
10	**42,991**	**48.85**	**67,520**	17,683	24.03	13,663	**42,194**	**23.63**	**32,056**	16,287	17.12	8,966

These "resources" are historical in nature and should not be relied upon. In 2004, they were compliant with applicable NI 43-101 criteria and CIM Standards and Definitions. Since 2004, more drilling has been added and more geological information has become available. Additionally, assumptions used to determine cut-off grades are likely to have changed since 2004. Consequently, these "resources" cannot be considered as current. They are included in this section for illustrative purposes only and should not be disclosed out of context.

In September 2004, a second milling test was conducted in the Camflo Mill facility and supervised by Edmond St-Jean, P.Eng. from Laboratoire LTM Inc., in Val-d'Or (St-Jean, 2004). A total of 9,005 short tons (8,169.4 metric tons) of underground ore from the Discovery Zone was milled. The high-grade ore represents 6,354 short tons (5,764.4 metric tons) grading some 0.362 oz/st (12.41 g/t Au). The low grade ore represents some 2,651 short tons (2,405.0 metric tons) grading 0.148 oz/st (5.07 g/t Au). Four bricks were casted, and each brick was marked and weighed. After casting the last brick, Camflo Mill personnel recovered a 921.9-gram button, and after cleaning the furnace, Camflo Mill personnel recovered a 207.1-gram button. The four bricks weighed 3,427.6 ounces in total. This total did not take into account the amount of gold in the matte and rich slag, or what was recovered after cleaning the tank house, because they were not analyzed. It was probable that they contain several ounces of gold (from 5 to 10 oz). The gold pour produced 3,500 ounces of doré containing 2,595.5 ounces of gold.

A mill malfunction occurred on September 11 when pressure in the presses increased abnormally. The presses were shaken in the evening by insufflating pressurized air into them. The color test showed signs of gold loss over a period of six hours during that night, but the situation had gone back to normal. According to St-Jean (2004), the quantity of gold lost to the wastes during the mill malfunction resulted in the loss of about 90 ounces of gold, which would normally be recoverable. For the total of 9,005 short tons (8,169.4 metric tons) the mill feed grade was estimated at 0.299 oz/st (10.25 g/t Au), with a recovery of 95.5%. After the final inventory of the mill, the grade was calculated at 0.312 oz/st (10.70 g/t Au), including gold lost in the tails during the milling. If the 90 ounces lost to the mill malfunction is included in the mill reconciliation, total gold recovery is close to 97%.

In November 2004, the FAJV was shut down due to legal action brought against Taurus by Fairstar and pending additional financing. On November 23, 2004, Taurus announced that it had agreed to combine with American Bonanza Gold Mining Corporation (“Bonanza”) to create a new gold company. Pursuant to the business combination, the new company also agreed to acquire Fairstar’s 38% interest in the Fenelon Gold Project.

6.11 2005–2008 Exploration Program (Bonanza)

In January 2005, InnovExplo published a 43-101 compliant technical report on the FAJV (Pelletier and Gagnon, 2005). This technical report contained a revised resource estimate of the Discovery Zone, which took into account the material removed during the 2004 bulk sampling program. Total resources were estimated at 55,684 tonnes grading 19.61 g/t Au in the measured and indicated categories (4,002 t at 18.36 g/t Au for measured, and 51,682 t at 19.71 g/t Au for indicated). This represented 35,107 ounces of gold. In addition, inferred resources were estimated at 27,245 tonnes grading 12.79 g/t Au, for a total gold content of 11,203 ounces. Of the combined measured and indicated resources, 7,757 tonnes had been removed by mining, which means the remaining total of measured and indicated resources were 47,927 tonnes grading 19.61 g/t Au (including 3,098 t of ore broken on site). Inferred resources had not changed. Measured resources were not recalculated after new development material was sampled because the authors of the report concluded it would have only a minor impact on grade and tonnage, but that a new estimate would have to be calculated following any future diamond drilling program.

These “resources” are historical in nature and should not be relied upon. In 2005, they were compliant with NI 43-101 criteria and CIM Standards and Definitions applicable at the time. Since 2005, more drilling has been added and more geological information has become available. Additionally, assumptions used to determine cut-off grades are likely

to have changed since 2005. Consequently, these "resources" cannot be considered as current. They are included in this section for illustrative purposes only and should not be disclosed out of context.

In 2005, InnovExplo also performed an exhaustive relogging and drill core sampling program (economic and whole-rock analyses) on the Discovery Zone deposit (local-scale) and the rest of the Fenelon Mine Property (property-scale) (Théberge et al., 2006). The drill core review, studies and sampling program mostly took place from September to mid-November 2005. The core from seventy-four (74) drill holes was reviewed, amounting to 7,895 metres within the Discovery Zone area, including 249 whole-rock geochemistry samples and 139 mineralized samples. The core from thirty-six (36) drill holes located outside of the Discovery Zone area, totalling 9,581 metres, was also reviewed, including 167 whole-rock geochemistry samples and 34 mineralized samples. The results of the geological review and sampling were combined with geophysical survey data (Mag, EM, and IP) and incorporated into MapInfo (GIS database) at the property-scale in order to completely revise the surface geological map of the Fenelon "A" Property (lithologies, favourable areas, faults and fold structures).

A drilling and sampling program was carried out from December 2005 to mid-April 2006 (Brousseau et al., 2007). A total of fifty-four (54) NQ-size diamond drill holes were logged and sampled for 18,113.9 metres on the Fenelon "A" Property, corresponding to thirty-three (33) diamond drill holes on the Discovery Zone and its extensions (east and west), and twenty-one (21) on the regional component of the drilling program, outside of the Discovery Zone area. This program included 359 whole-rock geochemistry samples and 2,837 mineralized samples.

In addition to the classic lithogeochemical description, a detailed geochemical and alteration study of the whole-rock geochemistry assays was produced by Mathieu Piché, an independent consultant working under the supervision of InnovExplo. The results of geological observations and the interpretation of alteration from that specific study were incorporated into MapInfo (GIS database) to review the mineral potential of the Discovery Zone area and the Fenelon Felsic Volcanic Complex (FFVC; Le Grand, 2008).

Bonanza carried out a two-phase diamond drilling exploration program on the Fenelon "A" Property during the winter of 2006–2007. The first phase comprised 959.20 metres in four drill holes drilled from December 5 to December 16, 2006, on the Discovery West Zone, which was known to carry gold. The second phase was carried out in the FFVC, comprising six (6) deep holes (>490 m) for a total length of 3,399.40 metres. This phase started on January 6, 2007, and was stopped on April 1, 2007 due to ground thaw. This drilling campaign focused on the new nickel mineralization in the northeastern part of the Property. This sector was also investigated for gold and massive sulphides.

The 2008 exploration program was planned for 2,500 metres of NQ-caliber drilling, however only one (1) hole was completed, reaching a depth of 349 metres in the FFVC area (Leclerc and Giguère, 2010). Another hole was abandoned.

6.12 2010–2011 Exploration Program (Balmoral)

On September 7, 2010, Bonanza and Balmoral announced that they had entered into a Letter of Offer whereby Balmoral was granted the exclusive right to acquire Bonanza's rights, titles and interests in a series of properties located in Québec and Ontario, including their Fenelon Property.

In late January 2011, Balmoral launched a diamond drill program targeting the Discovery Zone and its extensions. Forty-one (41) diamond drill holes were drilled totalling 8,579.9 metres (see press releases of Balmoral). Balmoral completed thirty-five (35) holes testing the lateral and down-dip/plunge extensions of the Discovery Zone. Results were highlighted by several very high grade gold intercepts that confirmed the high-grade tenor of the Discovery Zone. Drilling successfully extended a number of the mineralized veins comprising the zone along strike and to a vertical depth of 250 metres. The six (6) other holes were drilled farther to the east and north of the Discovery Zone.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 The Abitibi Terrane (Abitibi Subprovince)

The Fenelon Mine Property is located in the northwestern Archean Abitibi Subprovince in the southern Superior Province of the Canadian Shield. The Abitibi Greenstone Belt has been historically subdivided into northern and southern volcanic zones defined using stratigraphic and structural criteria (Dimroth et al., 1982; Ludden et al., 1986; Chown et al., 1992), mainly based on an allochthonous greenstone belt model development (i.e., interpreting the belt as a collage of unrelated fragments). The first geochronologically constrained stratigraphic and/or lithotectonic map (Fig. 7.1), interpreted by Thurston et al. (2008), includes the entire Abitibi Greenstone Belt known coverage span (i.e., from the western Kapuskasing Structural Zone to the eastern Grenville Province). Thurston et al. (2008) described the Abitibi Greenstone Belt as mainly composed of volcanic units that were unconformably overlain by large sedimentary Timiskaming-style assemblages. Similarly, both new mapping surveys and new geochronological data indicate an autochthonous origin for the Abitibi Greenstone Belt.

Generally, the Abitibi Greenstone Belt comprises east-trending synclines containing volcanic rocks and intervening domes cored by synvolcanic and/or syntectonic plutonic rocks (gabbro-diorite, tonalite and granite) alternating with east-trending turbiditic wacke bands (MERQ-OGS, 1984; Ayer et al., 2002a; Daigneault et al., 2004; Goutier and Melançon, 2007). Normally, the volcanic and sedimentary strata dip vertically and are usually separated by abrupt, variably dipping east-trending faults. Some of these faults, such as the Porcupine-Destor Fault, display evidence of overprinting deformation events including early thrusting, later strike-slip and extension events (Goutier, 1997; Benn and Peschler, 2005; Bateman et al., 2008). Two ages of unconformable successor basins are observed: a) widely distributed fine-grained clastic rocks in early Porcupine-style basins; followed by b) Timiskaming-style basins composed of coarser clastic sediments and minor volcanic rocks, largely proximal to major strike-slip faults, such as the Porcupine-Destor and Larder Lake-Cadillac faults and other similar regional faults in the northern Abitibi Greenstone Belt (Ayer et al., 2002a; Goutier and Melançon, 2007). The Abitibi Greenstone Belt is intruded by numerous late-tectonic plutons composed mainly of syenite, gabbro and granite with fewer lamprophyre and carbonatite dykes. Commonly, the metamorphic grade in the Abitibi Greenstone Belt varies from the greenschist to subgreenschist facies (Jolly, 1978; Powell et al., 1993; Dimroth et al., 1983; Benn et al., 1994) except in the vicinity of most plutons where the metamorphic grade corresponds mainly to the amphibolite facies (Jolly, 1978).

7.2 New Abitibi Greenstone Belt Subdivisions

As mentioned in section 7.1, new Abitibi Greenstone Belt subdivisions were defined using new mapping and geochronological data from the Ontario Geological Survey and Géologie Québec. The following section presents a more detailed description of these new subdivisions, mostly abridged from Thurston et al. (2008) and references therein.

Seven (7) discrete volcanic stratigraphic episodes define the new Abitibi Greenstone Belt subdivisions based on numerous U-Pb zircon age groupings. The new U-Pb zircon ages clearly show timing similarities for volcanic episodes and plutonic activity ages between the northern and southern portions of the Abitibi Greenstone Belt, as indicated in Figure 7.1. These seven volcanic episodes (Fig. 7.1) are listed below, chronologically from the oldest to the youngest:

- Volcanic episode 1 (pre-2750 Ma);
- Pacaud Assemblage (2750–2735 Ma);
- Deloro Assemblage (2734–2724 Ma);
- Stoughton-Roquemaure Assemblage (2723–2720 Ma);
- Kidd-Munro Assemblage (2719–2711 Ma);
- Tisdale Assemblage (2710–2704 Ma);
- Blake River Assemblage (2704–2695 Ma);

The Abitibi Greenstone Belt successor basins are of two types: 1) laterally extensive basins corresponding to the Porcupine Assemblage with early turbidite-dominated units (Ayer et al., 2002a); followed by 2) the aerially more restricted alluvial-fluvial or Timiskaming-style basins (Thurston and Chivers, 1990).

The geographic limit (Fig. 7.1) between the northern and southern parts of the Abitibi Greenstone Belt has no tectonic significance but is similar to the limits between the internal and external zones of Dimroth et al. (1982) and those between the Central Granite-Gneiss and the Southern Volcanic zones of Ludden et al. (1986). The boundary between the northern and southern parts passes south of the wackes of the Chicobi and Scapa groups with a maximum depositional age of 2698.8 ± 2.4 Ma (Ayer et al., 1998, 2002b).

The Abitibi Subprovince is bounded to the south by the Larder Lake-Cadillac Fault Zone, a major crustal structure that separates the Abitibi and Pontiac subprovinces (Fig. 7.1) (Chown et al., 1992; Mueller et al., 1996; Daigneault et al., 2002, Thurston et al., 2008).

The Abitibi Subprovince is bounded to the north by the Opatica Subprovince (Fig. 7.1), a complex plutonic-gneiss belt formed between 2800 and 2702 Ma (Sawyer and Benn, 1993; Davis et al. 1995). It is mainly composed of strongly deformed and locally migmatized, tonalitic gneisses and granitoid rocks (Davis et al., 1995).

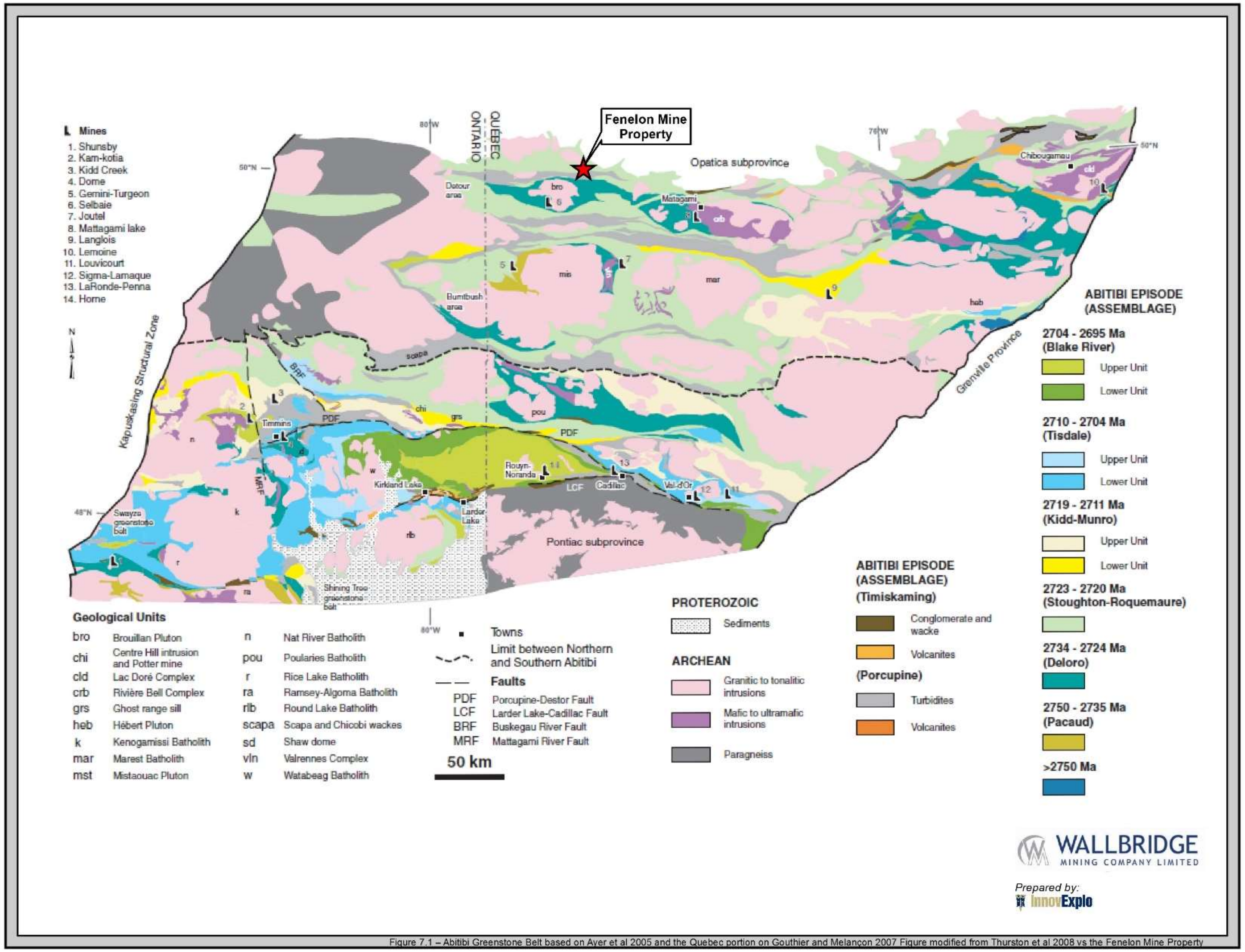


Figure 7.1 – Abitibi Greenstone Belt based on Ayer et al. (2005) and the Québec portion on Goutier and Melançon (2007). Figure modified from Thurston et al. (2008).

7.3 Regional Geology

The geology in the northwestern Abitibi Subprovince has been described by Lacroix et al. (1990), Ayer et al., (2002a) and Faure (2012, 2015), and is referred to the Harricana-Turgeon volcano-sedimentary segment. The segment extends from the Detour Lake mine (Ontario) in the west to Matagami (Québec) in the east, and includes the Matagami, Brouillan, Joutel and Casa-Berardi mining districts.

The segment is dominated by mafic volcanic rocks, followed by sedimentary and plutonic rocks. It is transected by numerous E-W trending deformation zones located either at the contacts of volcano-sedimentary units and granitoid plutons or crosscutting them (Fig. 7.2). The two major northernmost faults of the Abitibi are the Sunday Lake (SLDZ) and Grasset (GDZ) deformation zones (Fig. 7.2). The GDZ is the equivalent of the South Detour Deformation Zone in Ontario.

The main rock assemblage north of the SLDZ consists of tholeiitic basalts of the Manthet Group dated in Ontario, north of the Detour Lake mine, at 2722 Ma (Marmont and Corfu, 1989). The basalt sequence is dominated by pillowed and massive flows and is intruded by mafic and ultramafic sills and dykes. This group is the equivalent of the Stoughton-Roquemaure assemblage in Ontario, which has been dated between 2723 and 2720 Ma (Thurston et al. 2008).

The volcanic package south of the GDZ is attributed to the Brouillan-Fenelon domain (Lacroix et al., 1990) and is subdivided in two volcanic assemblages. The older assemblage consists of bimodal andesite-rhyolite calc-alkaline volcanism and magmatism dated between 2725-2730 Ma and is correlated to the Deloro in southern Abitibi (Barrie and Krog, 1996; Thurston et al. 2008). This package of volcanic rocks is flanked around the Brouillan synvolcanic pluton and in the core of the Brouillan anticline, and hosts the Selbaie polymetallic epithermal deposit (Faure et al., 1996). The felsic volcanic rocks that host the volcanogenic massif sulphides deposits in the Matagami mining camp are also attributed to this package. The mafic assemblage south of the GDZ has similar volcanic facies and composition to the Manthet group with few ultramafic complexes and is correlated to Stoughton-Roquemaure assemblage.

Metasediments are present in two different rock packages. The synorogenic flysch-type sediments of the Matagami assemblage is wedged between the Sunday Lake and the Grasset deformation zones. The Matagami sediments are composed of interbedded argillaceous siltstones and wackes (turbidites sequences) and minor mafic to felsic volcaniclastic rocks. They are interpreted to be formed in a successor basin unconformably overlying the volcanic rocks (Mueller et Donaldson, 1992). They are equivalent in Ontario to the Caopatina sediments (2698 Ma) and to a broader scale to the Porcupine-type sediments in the southern Abitibi. A large basin of polygenic conglomerates, 15 kilometres long by 2.5 kilometres wide, occurs in the center of the segment north of the SLDZ. This late restricted basin is bounded by faults and has the hallmarks of Timiskaming-style divergent fault-wedge basin, a variant of a pull-apart basin, developed proximal to major strike-slip faults in southern Abitibi (Mueller et al., 1991). A similar conglomeratic basin occurs along the South Detour Fault in Ontario (e.g. extension of the Grasset fault). These conglomeratic basins are spatially associated with orogenic and syenite gold deposits elsewhere in the Abitibi (Robert, 2001). A few layers of sulphidic and graphitic shale or tuffs (tens to hundreds of metres), highly conductive, are interlayered between basaltic flows or within the Matagami sediments.

Apart from the gabbro and ultramafic sills and dykes, the plutons in the NW Abitibi are felsic to intermediate in composition. Three major intrusions are present; the Brouillan, Jérémie and Turgeon. The Brouillan Pluton is a polyphase mafic tholeiitic to felsic calc-alkaline synvolcanic intrusion dated at 2729 Ma (Barrie and Krogh, 1996). The Jérémie and Turgeon plutons, as well as smaller granodiorite and diorite intrusions, have metamorphic aureoles reaching upper greenschist to lower amphibolite facies, and they are interpreted as pre- to syn-kinematic (Lacroix, 1994).

The rock sequence has been affected by regional deformation and metamorphism. The metamorphism increases towards the Opatica Subprovince, from greenschist facies in the south to the amphibolite to the north. The appearance of the hornblende that marks the amphibolite isograd occurs between 2 to 5 kilometres south of the limit between the two subprovinces (Lacroix, 1994).

The sparse stratification measurements recorded north of the SLDZ indicate that the dip of the basalt flow sequence is moderate to steep. Fold patterns have been interpreted based mainly on the distribution of magnetic highs corresponding to gabbroic and ultramafic sills, and electromagnetic conductors that characterize graphitic tuffs and sediment horizons. The folds are inclined and open to tight, with axial traces oriented NW-SE, except around the Detour Lake mine and north of the Jérémie Pluton where they are isoclinal.

The SLDZ and the GDZ are the major structural features in the area. They can be traced over 150 kilometres from the western boundary of the Abitibi Subprovince in Ontario to the east of the Fenelon Mine Property and to the north of the Matagami mining camp (Fig. 7.2). These two faults share many characteristics with other major breaks of the Abitibi in that they are wide corridors of ductile and high-strain deformation with a mixture of highly altered volcanic, sedimentary and intrusive rocks, including ultramafic slices and syn-orogenic felsic to intermediate dykes. At the Detour Lake mine, the SLDZ displays overprinting deformation events, including early thrusting with later sinistral and dextral strike-slip events (Oliver et al., 2012). On the regional map of total magnetic field, the fault is defined as a linear east-west-trending magnetic low that truncates, at a high angle, domains of rock units with low and high magnetic signatures to the north and the less contrasting magnetic signatures of sediments to the south.

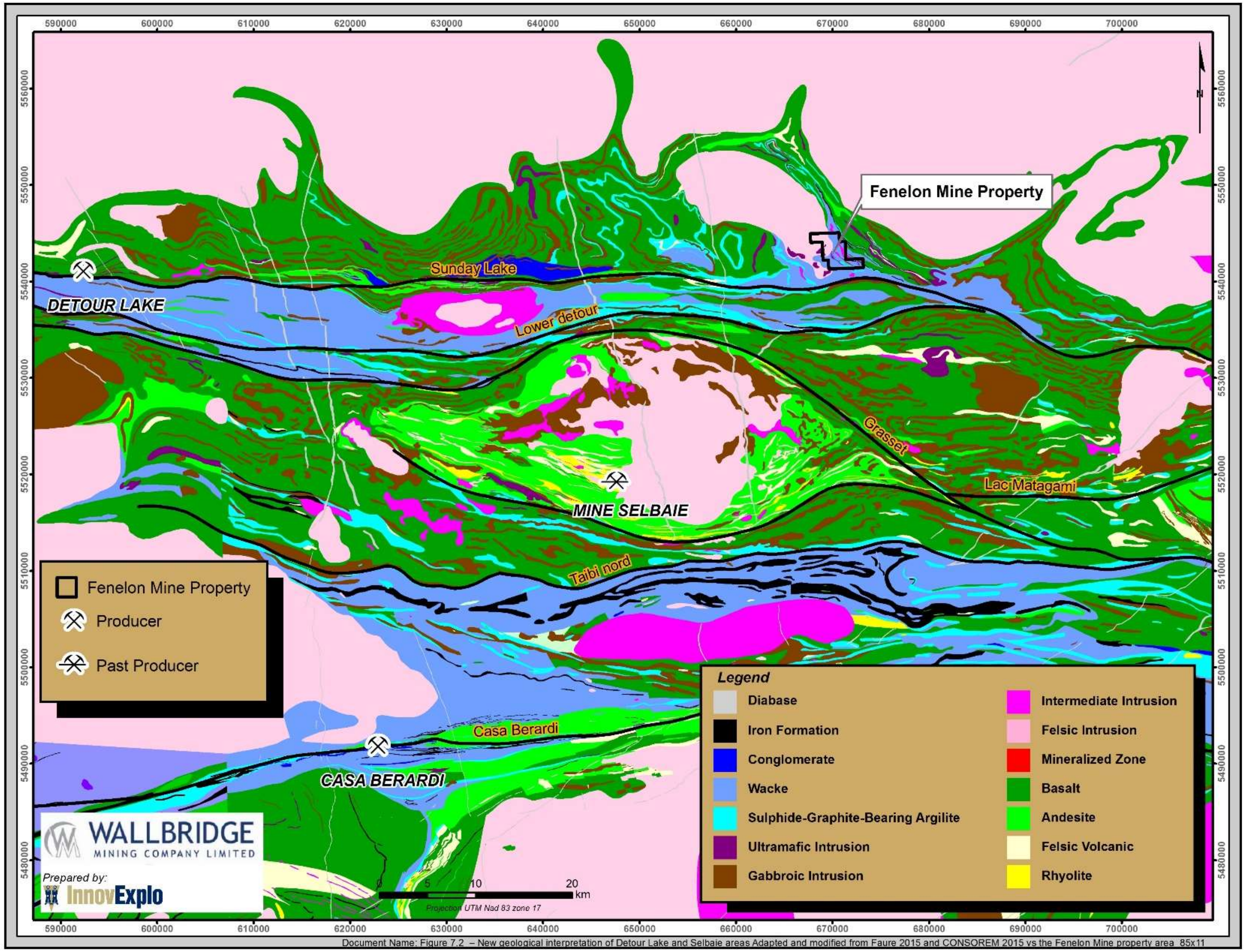


Figure 7.2 – New geological interpretation of the Detour Lake and Selbaie areas. Adapted and modified from Faure (2015) and CONSOREM (2015).

7.4 Fenelon Mine Property Geology

The following description of property geology was taken from the technical report produced by Pelletier and Gagnon (2005) and retains the references therein.

The Fenelon Mine Property is covered by 4 to 50 metres of glacial overburden consisting mainly of sandy and gravel outwash material and lesser boulder-rich tills. There are no natural rock outcrops in the area of the Discovery Zone (a.k.a. the Fenelon deposit) where glacial overburden is generally 4 to 8 metres thick. Detailed property-scale geological information is only available for this area, which has been drilled and bedrock exposures created during open pit sampling and underground development work. The correlation between geological information and geophysical maps has contributed to the recognition of certain magnetic units such as gabbroic and ultramafic rocks, low magnetic sedimentary rocks and highly conductive graphitic horizons (Lacroix, 1994; Faure, 2012, 2015).

The Fenelon Mine Property is situated near the SLDZ, with the Discovery Zone located along a northwest-trending splay roughly two kilometres north of the east-west trending SLDZ. In the vicinity of the Discovery Zone, the SLDZ warps gently to the south to strike ESE immediately east of the claim block. Ground and airborne geophysical data suggest that several splay structure systems extend northward from the SLDZ into the Discovery Zone area. The absence of outcrop exposure in the area impedes the ability to accurately map fold patterns. However, regional airborne geophysical data suggest that rock units are folded. According to Lacroix (1991), the Discovery Zone area may be located within a regional antiformal structure with an axial trace trending NW through the core of the Jérémie Pluton. Airborne magnetic data also suggests the presence of several more brittle faults and/or shear zones striking E, NNW and NE. Such structures are outlined by sharp breaks and displacements of magnetic markers. In 1997, a drilling program provided sporadic oriented core (Foster testing) on the Discovery Zone. According to Pelletier and Gagnon (2005), the interpretation of this oriented core data demonstrates that within the drilling area, the dominant planar fabric strikes E to ESE with a steep southerly to vertical dip (70°–90°). However, given the lack of lateral deviation data for the 1997 drilling program, the interpretation of “Foster test” results is equivocal.

The Manthet Group, located north of the SLDZ, underlies the entire Property. Although published geological maps (Lacroix, 1991) indicate that the Property should be underlain by basaltic volcanic rocks of the Manthet Group, diamond drilling over the Property suggests that the geology is predominantly characterized by dominantly mafic volcanic rocks and pelagic sedimentary rocks, with a smaller amount of felsic to intermediate volcanic rocks and tuffs, and ultramafic volcanic rocks. Small intrusions and synvolcanic to pre-tectonic dykes, mostly mafic to intermediate, are documented in volcanic and sedimentary succession. The Jérémie Pluton, a large plutonic body of intermediate to felsic (diorite–tonalite–granodiorite), syn- to late-tectonic units, occurs at a few kilometres northwest of the Discovery Zone. In drill logs and reports, lithological units are described as variably altered, and the dominant alteration types include silicification, carbonatization, sericitization, biotization, chloritization and the addition of sulphides. Mafic to ultramafic intrusive units are locally magnetic.

7.5 Discovery Zone Geology

The following description of the Discovery Zone geology is taken from the technical report produced by Pelletier and Gagnon (2005), and retains the references therein.

The Discovery Zone is hosted in a series of siliceous zones and small-scale silica-albite shear zones within coarse-grained mafic intrusives that are segmented by a series of mafic dykes, between two panels of argillaceous sediments.

7.5.1 Lithology

The Discovery Zone area is characterized by four major lithological units. The dominant unit is metasedimentary and comprises greywackes, siltstones, mudstones, locally graphitic argillites and iron formations.

A major mafic intrusive unit intrudes the metasediments. This gabbroic unit is dark-coloured, massive and usually coarse grained (1–4 mm), although it is locally medium grained as seen south of the ramp (Fig. 7.3).

The second type of intrusive unit to cut the metasediments is intermediate to felsic. This unit is located north of the main mafic intrusive where it displays massive texture. It is generally medium grained and locally porphyritic with feldspar. In the decline ramp, this unit occurs as a swarm of narrow feldspar porphyry dykes (centimetric to decametric) in sharp contact with the metasediments (Fig. 7.3). The third type of intrusive rock is represented by late mafic, fine-grained dykes. They range from a few centimetres wide to 2–3 metres, and locally cut the mineralized zones, creating internal dilution.

Pelletier and Gagnon (2005) examined the outcrop stripped along the southeast extension of the small open pit excavated in 2001, as well as all the underground development. Critical relative timing relationships between lithological units, deformation, alteration and gold mineralization were exposed in these locations. The description of the stripped outcrop was based in part on Couture and Michaud (2003). The stripped outcrop and the underground development exposed a sequence of steeply-dipping deformed layered rocks consisting of alternating fine-grained argillaceous sedimentary rocks, greywackes and felsic siliceous rocks, crosscut by a major massive, coarse-grained mafic intrusion. both units are crosscut by a plethora of fine-grained mafic dykes (Figs. 7.4 and 7.5). The feldspar porphyry dykes clearly cut the sediments, but the relationship with the coarse-grained mafic intrusives is not well exposed. The subvertical layering in the rock units trends approximately SE and was overprinted by a roughly subparallel penetrative foliation fabric.

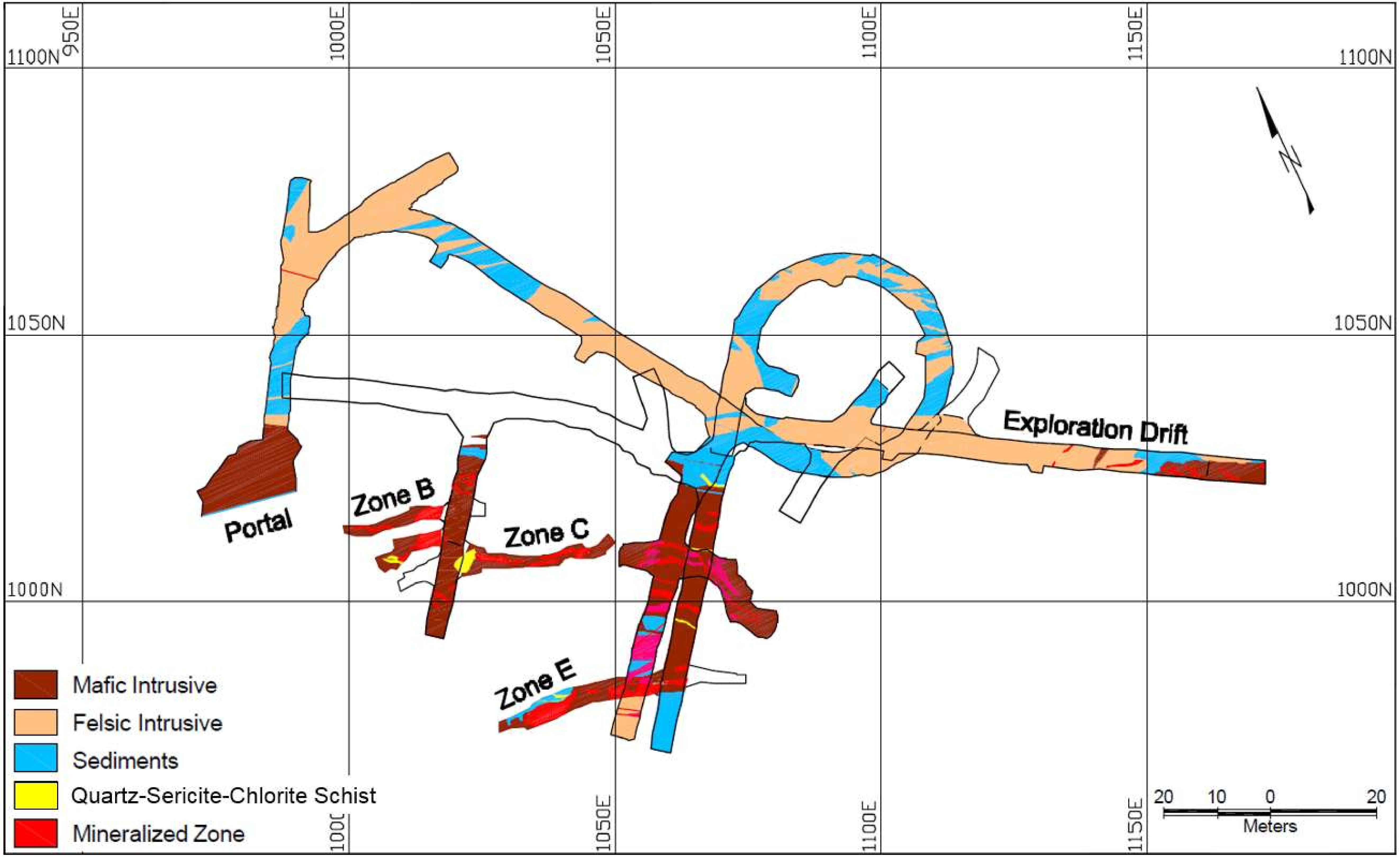


Figure 7.3 – Geological mapping of underground workings on the Discovery Zone (from Pelletier and Gagnon, 2005)

Figure 7.4 – Sharp contact between a late fine-grained mafic dyke (upper part of the photo) and the coarse-grained mafic intrusion (lower part of the photo). Photo from Pelletier and Gagnon (2005)

Figure 7.5 – North-south-trending late fine-grained mafic dyke crosscutting the southeast-trending sediments. Photo from Pelletier and Gagnon (2005)

The stripped outcrop was subdivided by Pelletier and Gagnon (2005) into three areas approximately perpendicular to layering. The northeastern portion of the outcrop consists chiefly of argillaceous and greywacke sedimentary units cut by narrow (<1 m) highly deformed mafic dykes (Fig. 7.5). The southwestern portion of the stripped outcrop is occupied by a massive black silica rock, a mottled silica breccia and two feldspar porphyry dykes, all injected by numerous deformed narrow mafic dykes, less than 1 centimetre to a few metres in thickness. The origin of the massive silica rock is not known. The central portion of the outcrop, which hosts most of the gold mineralization, is occupied by a mafic dyke complex that appeared to be injected along the contact between the intermediate to felsic silica rock and the layered sedimentary sequence. The mafic dyke complex consists of a thicker coarse-grained massive mafic dyke injected by numerous thinner (< 1 m) parallel mafic dykes (Fig. 7.6). In section, the dyke swarm dips steeply (75°–80°) to the south. Couture and Michaud (2003) observe that the thicker massive dyke was weakly strained and, locally near the pit wall, an intrusive breccia developed. This breccia and the crosscutting relationships between narrow dykes indicate repetitive dyke events. On either side of the sheeted mafic dyke swarm, narrow highly folded mafic dykes extend into surrounding lithologies. The origin of the black silica rock to the southwest of the dyke complex remains enigmatic. This rock is very massive and fine grained.

One feldspar porphyry dyke occurs between the central dyke swarm complex and the mottled silica breccia rock. It is in sharp intrusive contact with the massive black silica rock. Contacts relationships with the mottled silica rock and mafic dykes are, however, equivocal. The feldspar porphyry dyke is fairly massive and contains abundant centimetre-scale xenoliths. It is foliated and cut by several narrow mafic dykes. Laminated albite-quartz veins occur in the mottled silica breccia and massive black silica rock on either sides of the feldspar porphyry dyke. In the massive black silica rock, the veins are regular but severely buckled. In the mottled silica breccia, the veins are strongly boudinaged and also occur as angular to rounded clasts floating in the silica breccia. Folded and boudinaged veins locally contain sulphides (pyrrhotite, pyrite, ±chalcopyrite). It is suggested that these veins were related to the porphyry dyke. The crosscutting relationship between the albite-quartz veins and the massive black silica rock, along with their severe deformation in the mottled silica breccia, suggest the veins and porphyry dyke intruded the massive silica rock and were possibly coeval with silica breccia development. Porphyry dyke intrusion, albite-quartz-sulphide veins, silica breccia and sulphide stockwork clearly predate the intrusion of mafic dykes and also predate the development of the penetrative foliation.

The portal of the ramp is in the north wall of the open pit. The decline ramp passes underneath the pit, then cuts across the same lithological units observed at surface, thereby providing a better three-dimensional understanding of the units and structures. The decline ramp is passes through the sediments that occur to the north of the main coarse-grained intrusive unit (Fig. 7.3). This portion of the sediments was intruded by a swarm of feldspar porphyry dykes, and numerous sharp intrusive contacts are observed (Fig. 7.3).

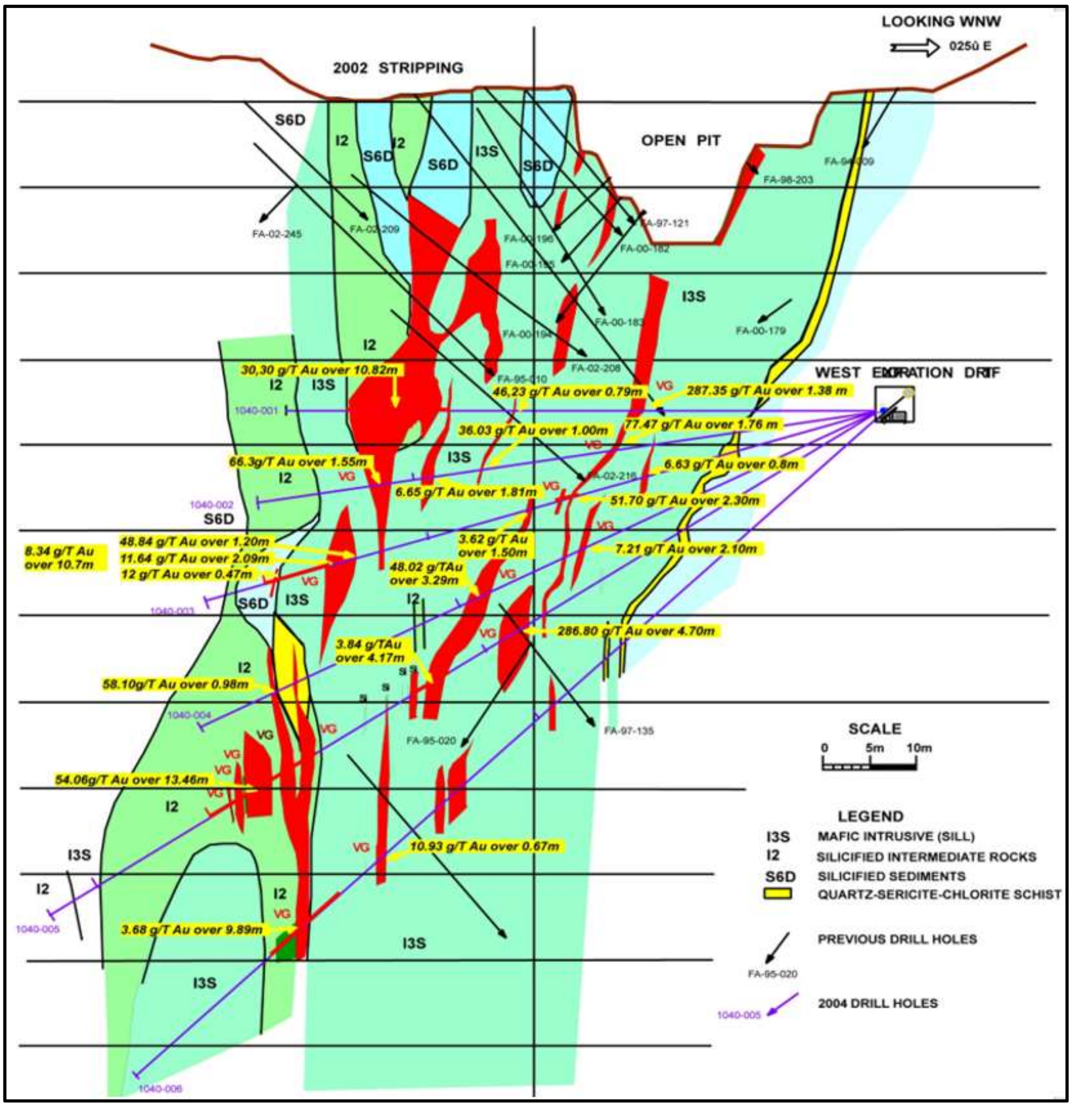


Figure 7.6 – Section 1040 E showing the mineralized zones (red color) and their host rocks (from Pelletier and Gagnon, 2005).

The eastern portion of the exploration drift on the 5213 sublevel was excavated in the intermediate to felsic, massive intrusive unit. The grain size was mainly medium and equigranular with some areas having a porphyritic texture. The end of this drift exposed the contact of the sediments with the coarse-grained mafic unit. No clear relationship between the intermediate intrusion and the coarse grain mafic intrusive was observed in this area. The western portion of the exploration drift was in the sediments. The three north-south crosscuts, one on the 5228 sublevel and two on the 5213 sublevel, were in the coarse-grained mafic unit, crosscut by some late fine-grained mafic dykes. The end of the crosscut TB-A on the 5213 sublevel was in the sediments. The three crosscuts intercepted the B-C and the D-E mineralized zones.

7.5.2 Structural Elements

Lithologies exposed on the stripped outcrop belong to a homoclinal volcanosedimentary panel intruded by a plethora of dykes. A stratigraphic top direction could not be readily determined in sedimentary units. Nonetheless, there is no structural evidence supporting the presence of large-scale folding at the Discovery Zone. All lithologies display a penetrative foliation and strain associated with this deformation is strongly partitioned throughout the outcrop. The southern and northern contacts of the mafic dyke swarm with argillaceous sediments exhibit wider zones of penetrative foliation. In the central corridor occupied by mafic sheeted dykes, strain is strongly partitioned into small-scale shear zones that have followed mafic dyke contacts.

Overall, the structural elements of both the wider deformation zones and small-scale shear zones are compatible with one phase of ductile deformation. Both small-scale and wider deformation zones display similar kinematics, with associated strongly developed stretching lineations and foliations. The stretching and mineral lineations observed at Fenelon are very strongly developed, indicating that a strong extension is associated with this deformation. Kinematic indicators, such as striated slip surfaces with hydrothermal steps and foliation/deformation zone orientations, support a south-over-north reverse-dextral displacement along both the wider and smaller-scale deformation zones. Foliations strike consistently NW-SE, with an average orientation of 296°/89° (strike/dip); lineations consistently rake east in the plane of the foliation, with an average orientation of 110°/78° (trend/plunge). A compilation of structural data collected by Couture and Michaud (2003) in 2002 indicates that the fold and boudin axes are consistently subparallel to the stretching and mineral lineations observed at Discovery Zone. The orientation of foliations measured at the Discovery Zone is similar to the orientation of small scale-shear zones. Late shear fracture-hosted quartz veins have a similar strike to the foliation, but dip at 45° to the foliation. In short, all structural elements observed on the Fenelon Mine Property are consistent with a single progressive deformation event. It is strongly suggested that the penetrative foliation, the small scale folds and deformation zones and the late quartz veins all developed during a single progressive deformation event primarily involving compressive shortening, reverse dip-slip kinematics with a minor component of dextral slip.

7.6 Gold Mineralization in the Discovery Zone

The following description of gold mineralization in the Discovery Zone is taken from the technical report produced by Pelletier and Gagnon (2005), and retains the references therein.

The gold mineralization is associated with a corridor of intense alteration located close to the contact between sediments and the coarse-grained mafic intrusives and within the coarse-grained mafic intrusive. Silicification is the dominant alteration and appears to control the mineralization. Sericite, biotite and black chlorite are also associated with the mineralized zones, but these alterations are not as continuous as the silicification. Some observations show a good correlation between high-grade values and a local increase in black chlorite content. Silicification serves as a guideline for exploration and is the key feature in guiding underground development. The general orientation and dip of the silicified and mineralized envelopes is subparallel to the contact of the sediments and the coarse-grained mafic intrusives (Fig. 7.6). Local variations in the orientation and dip are present. The thickness of these envelopes varies from a few centimetres to 15 metres.

Gold mineralization is concentrated in the silicified envelopes and is associated with sulphides such as pyrrhotite, chalcopyrite and pyrite. Sulphides are mainly disseminated, although where silicification is locally more intense, they are contained in quartz veins (Fig. 7.7-A, B, D). Pyrrhotite is dominant and its abundance generally varies from trace amounts to 30%, with intersections of massive pyrrhotite over a few centimetres. Chalcopyrite content generally varies from trace amounts to 15%, locally up to 40%. When present, pyrite occurs as trace amounts or up to 2%. Marcasite has been observed in drill core at depth and is locally associated with gold mineralization. Native visible gold is fairly common in drill hole intersections and in the wall rocks of developments. The grain size of the visible gold can reach 4 millimetres (Fig. 7.7-C, D).

The mineralization described above occurs in two distinct styles and two distinct stages in the Discovery Zone, predominantly within a wide corridor delimited by the extent of the coarse-grained mafic intrusives. The mineralization styles are as follows:

- Style 1: Early massive, laminated or brecciated silica-sulphide zones occurring along mafic dyke contacts, or commonly as isolated, irregular, metre-scale lensoid bodies inside the mafic dyke complex, like xenoliths of mineralized zone in the coarse-grained mafic intrusion (Fig. 7.8). Pyrrhotite and pyrite are the dominant sulphides and occur as narrow fracture fillings or disseminations in silica-rich rock.

- Style 2: Late narrow, lenticular or commonly tabular zones of silica-sulphide sericite alteration associated with small-scale (1–30 cm) shear zones occurring primarily along narrow dyke contacts. Sulphides occur disseminated in the altered rock or in quartz veinlets. The dominant sulphides are pyrrhotite, pyrite and chalcopyrite, with local coarse visible gold (Fig. 7.9).

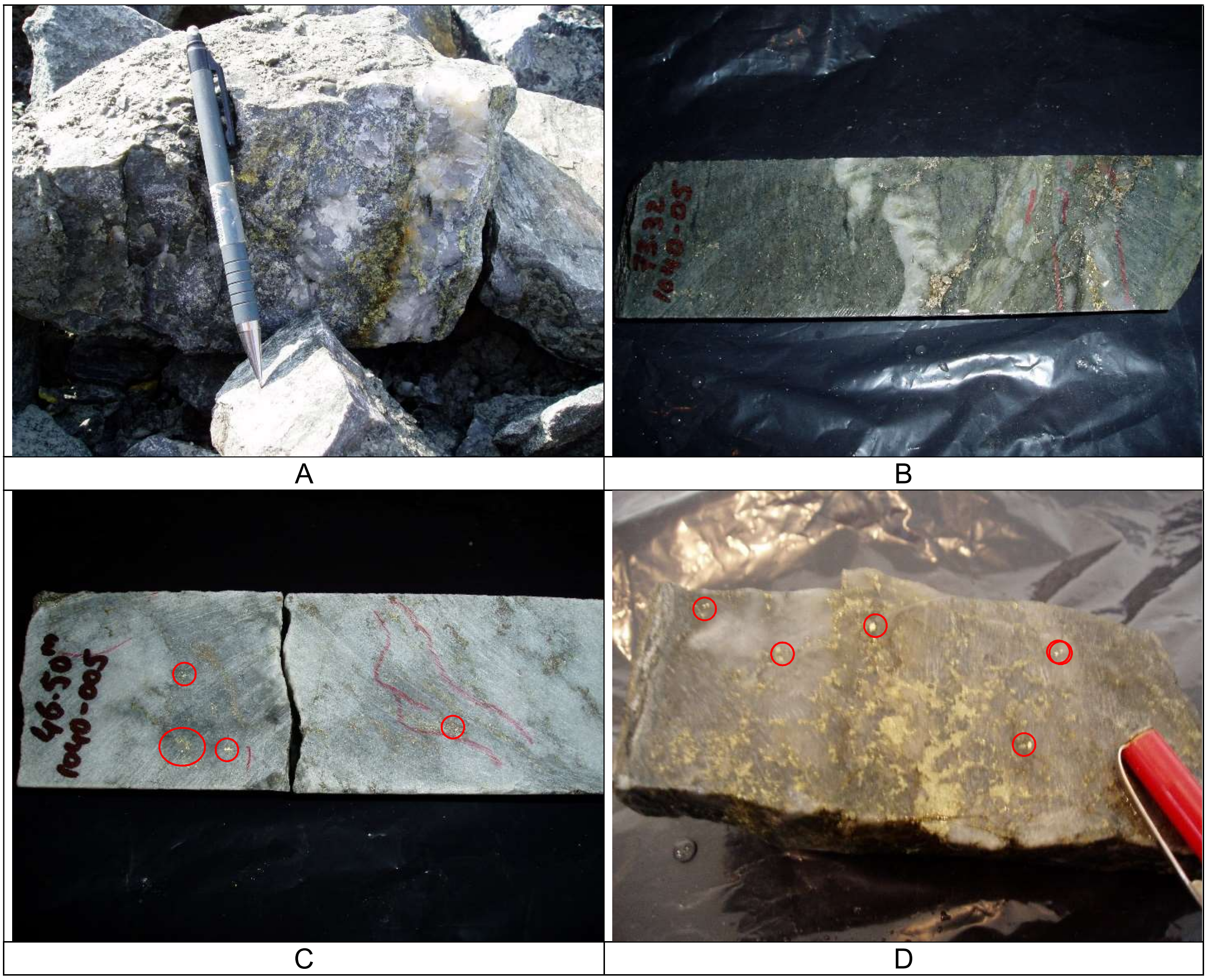


Figure 7.7 – A) Quartz veinlets with sulphides and disseminated sulphides in the wall rock. B) Hole 1040-005, 73.3 metres from collar: quartz veinlets with native coarse gold and disseminated sulphides in the wallrock. C) Hole 1040-005, 46.5 metres from collar: silicified zone with disseminated sulphides and native coarse gold. D) Rock from the stockpile of mineralized material: silicified zone with large amount of chalcopyrite and native coarse gold. (From Pelletier and Gagnon, 2005)

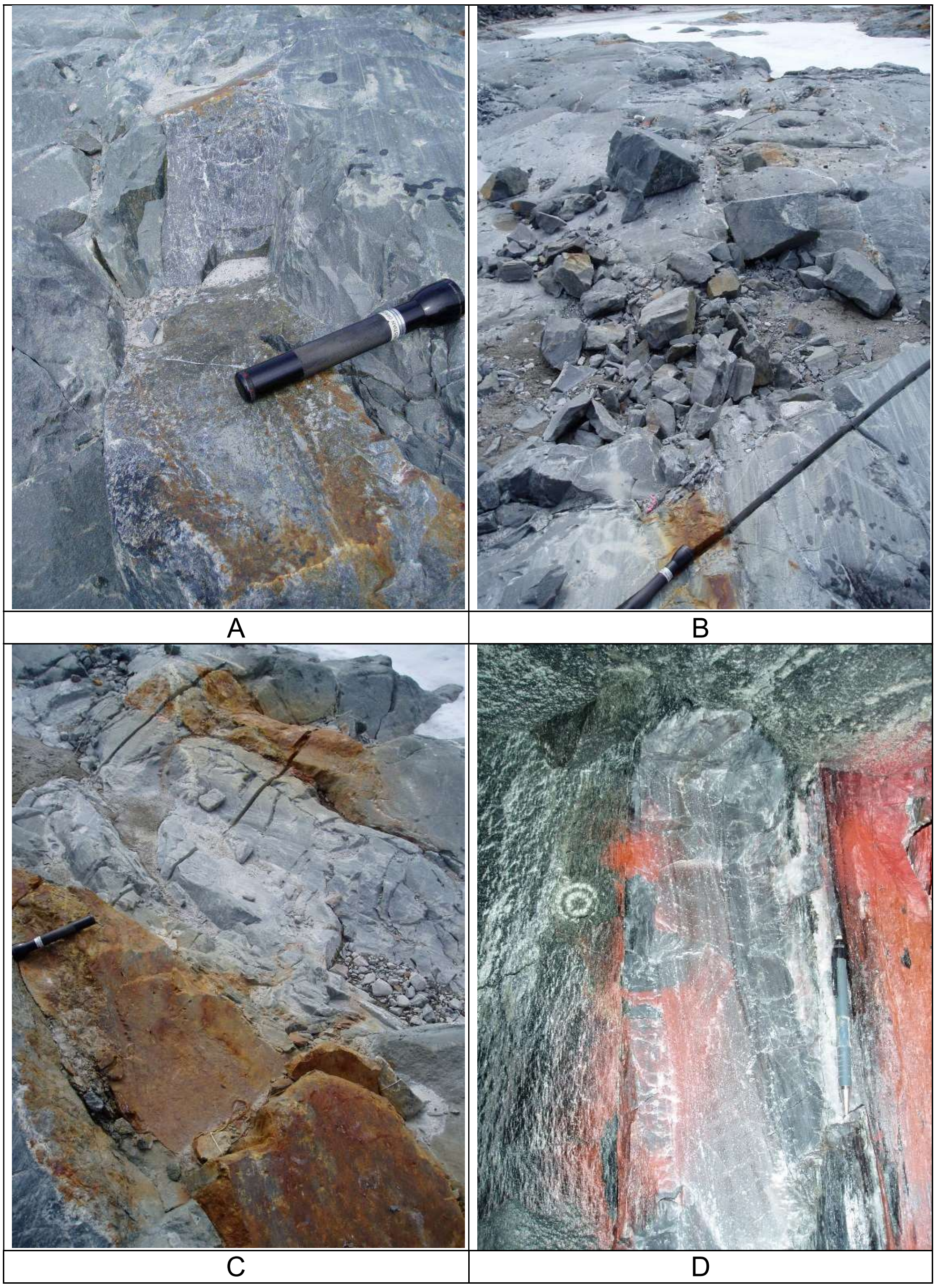
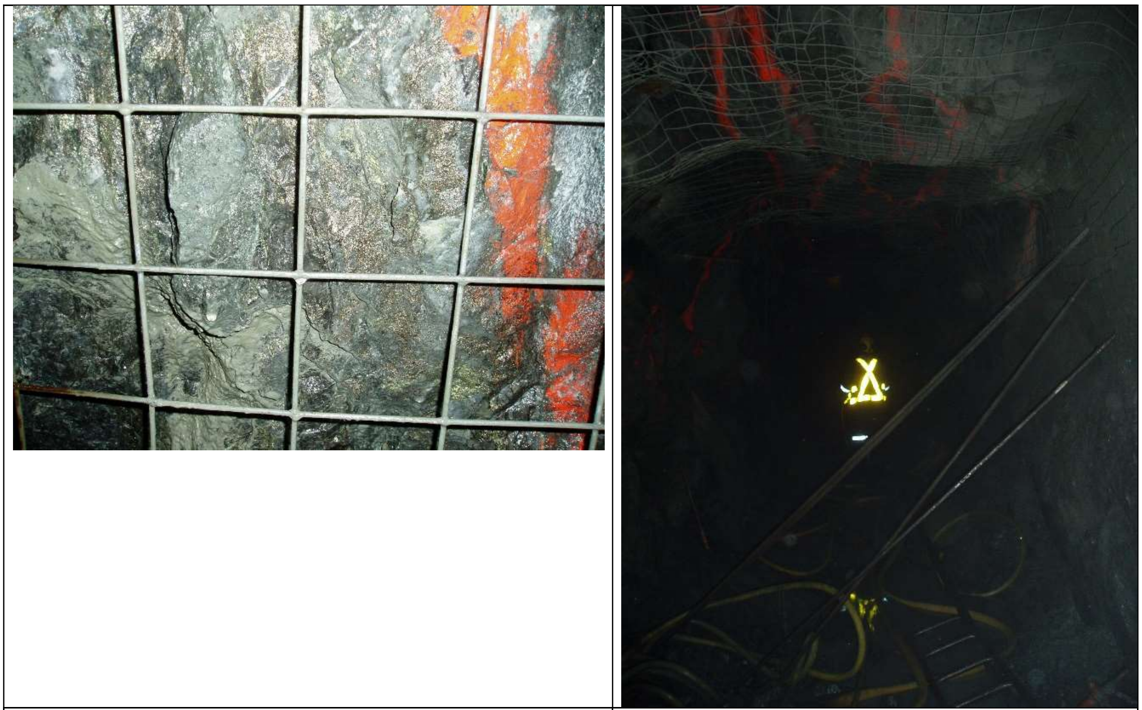


Figure 7.8 – Style 1 of mineralization: A) Lensoid body (xenolith) of early massive laminated silica-sulphide zones. B) Alignment of xenoliths along N115. C) Xenolith crosscut by a late mafic dyke. D) Xenolith in the coarse-grained mafic intrusive. (From Pelletier and Gagnon, 2005)



A	B

Figure 7.9 – Style 2 of mineralization: A) Disseminated pyrrhotite and chalcopyrite in the coarse-grained mafic intrusive associated with quartz veinlets. B) The same zone as in (A), but in the 5213 BC sublevel; the orange paint lines follow the zone, which is continuous for 40 metres (From Pelletier and Gagnon, 2005).

Crosscutting relationships clearly suggested that sulphide mineralization was emplaced during at least two distinct mineralizing episodes. Style 1 sulphide mineralization predated the coarse-grained mafic intrusive emplacement and predated penetrative deformation. The discontinuous distribution of these pods was interpreted to have resulted from the disruption of a previously continuous silica-sulphide layer or horizon by intrusion of coarse-grained mafic intrusives (Fig. 7.8). The second style of sulphide mineralization clearly postdated the coarse-grained mafic intrusive emplacement and predated the repeated intrusion of mafic dykes. It was associated with small-scale anastomosing shear zones commonly developed in the coarse-grained mafic intrusives and it was contemporaneous with the penetrative deformation.

8 DEPOSIT TYPE

8.1 Orogenic Gold

Metamorphic belts like the Abitibi are complex regions where accretion or collision has added to, or thickened, continental crust. Gold-rich deposits can be formed at all stages of orogen evolution, so that evolving metamorphic belts contain diverse gold deposit types that may be juxtaposed or overprint each other (Groves et al. 2003).

The majority of gold deposits in metamorphic terranes are located adjacent to first-order, deep-crustal fault zones (e.g., Cadillac-Larder Lake, Porcupine-Destor, Casa Berardi and Sunday Lake in the Abitibi), which show complex structural histories and may extend along strike for hundreds of kilometres with widths of as much as a few thousand metres (Goldfarb et al., 2005). Fluid expulsion from crustal metamorphic dehydration along such zones was driven by episodes of major pressure fluctuations during seismic events. Ores formed as simple to complex networks of gold-bearing, laminated quartz-carbonate fault-fill veins of second- and third-order shears and faults, particularly at jogs or changes in strike along the major deformation zones. Mineralization styles vary from stockworks and breccias in shallow, brittle regimes, through laminated crack-seal veins and sigmoidal vein arrays in brittle-ductile crustal regions, to replacement- and disseminated-type orebodies in deeper, ductile environments (Groves et al., 2003). Most orogenic gold deposits occur in greenschist facies rocks, but significant orebodies can be present in lower and higher grade rocks. The mineralization is syn- to late-deformation and typically post-peak metamorphism. They are typically associated with iron-carbonate alteration. Gold is largely confined to the quartz-carbonate vein network, but may also be present in significant amounts within iron-rich sulphidized wall-rock selvages or within silicified and sulphide-rich replacement zones (Dubé and Gosselin, 2007). One of the key structural factors for gold mineralization emplacement is the late strike–slip movement event that reactivated earlier-formed structures within the orogeny (Goldfarb et al., 2001), a condition that has been achieved along the SLDZ (Oliver et al., 2012).

In addition to the Discovery Zone, two significant gold occurrences are located along the SLDZ: the giant Detour Lake mine and the Bug Lake Trend. These gold occurrences present many similarities with mesothermal orogenic gold deposits in terms of metal associations, wall-rock alteration assemblages and structural controls.

8.1.1 Detour Lake Gold Mine

The geology of the Detour Lake gold mine has been studied in detail by Oliver et al. (2012) and Anwyll et al., 2016), and the principal characteristics of the ore zones are summarized here.

The Detour Lake area is comprised of a thick sequence of mafic to ultramafic volcanic rocks, referred to as the Deloro Assemblage (“DA”), in structural contact to the south with the younger sediments of Caopatina Assemblage (“CA”). This contact between the DA and CA is characterized by a regional-scale thrust zone referred to as the Sunday Lake Deformation Zone ("SLDZ").

The structures of the SLDZ are spatially related to most of the gold mineralization observed in the Detour Lake area. The gold mineralization in the Detour Lake area is believed to be relatively late and emplaced after tectonic juxtaposition of the DA and CA. At both Detour Lake and West Detour, gold mineralization is principally observed north of the SLDZ (hanging wall) along an east-west strike length of over 8 kilometres within a corridor several hundreds of metres wide. It forms a stockwork of auriferous quartz veins that splay from a flexure that coincides with the northern limb of a shallow west plunging antiform.

Two types of gold mineralization have been recognized:

- A wide and generally auriferous sulphide-poor quartz vein stockwork formed in the hanging wall of the SLDZ. The sulphide-poor quartz vein stockworks observed in the hanging wall have subvertical north or south dips and are parallel to a series of east-west trending high-strain zones. These veins form a weak stockwork and are boudinaged and/or folded.
- Gold mineralization that overprints the early auriferous stockwork, principally in the hanging wall of the SLDZ, with a higher sulphide content. The sulphide-rich gold mineralization predominantly fills structural sites in deformed quartz veins, fractures and veins crosscutting the foliation fabric, but also in pillow breccias and selvages. The distribution of sulphide-rich mineralization is strongly controlled by the geometry of kinematic orientation (i.e., pyrite and pyrrhotite concentrations have a shallow westerly plunge similar to the plunge of the main flexure zone in the SLDZ at an angle of about 40° in the area of the former open pit, shallowing to approximately 10° further to the west).

The gold mineralization occurs in different rock types within broad subvertical mineralized envelopes, and splits into several domains sub-parallel to the orientation of the SLDZ. It is principally contained in discrete networks of fault-fill or shear-hosted extensional quartz veins and broad, lithologically controlled mineralized zones with a weaker vein association.

As at December 31, 2015, the NI 43-101 Proven and Probable reserves for the Detour Lake mine were estimated at 445.5 Mt grading 1.01 g/t Au, for a total of 14.48 Moz of gold (Anwyll et al., 2016).

InnovExplo did not review the database, key assumptions, parameters or methods used by Anwyll et al. (2016) for the 2015 mineral reserve estimate. The reserve estimate was stated as compliant with NI 43-101 criteria by Anwyll et al. (2016), however InnovExplo is not able to confirm if new scientific or technical material information has become available since the effective date of the estimate. Consequently, InnovExplo cannot certify that the 2015 mineral reserve estimate is still complete and current.

8.1.2 Bug Lake Trend

Balmoral owns a 100% interest in the Martiniere Property, which hosts a number of near-surface occurrences of gold mineralization, including the West, Central and Bug Lake zones (or trends). More information about the Bug Lake Trend is presented in section 23.4 – *Martiniere Property*.

9 EXPLORATION

In 2016, Wallbridge initiated an exploration program on the Fenelon Mine Property. The first phase of the program involved a review of historical drilling in close proximity to the mine workings and additional sampling of previously unsampled historical drill core where warranted. The results from the first three batches of samples included a sample with visible gold and assayed 89.3 g/t Au over 0.35 metre. Assay results for an additional 124 samples are pending.

Results from the first two batches were announced in the issuer's press release of November 16, 2016. Of the 176 samples (179 metres), 25 (14%) returned values greater than 0.5 gram per tonne. Highlights included the following:

- 89.30 g/t Au over 0.35 m in hole 1050-005;
- 4.21 g/t Au over 0.72 m in hole 1100-001;
- 3.91 g/t Au over 0.99 m in hole 1110-001;
- 2.55 g/t Au over 1.57 m in hole FA-02-214.

Results from the third batch were announced in the issuer's press release of December 5, 2016. Of the 275 new samples, 3 samples returned values greater than 5 g/t, 29 samples (>10%) returned >0.5 g/t, and 34 samples returned grades ranging from 0.5 g/t to 0.1 g/t. Highlights included the following:

- 19.7 g/t gold over 1.90 metres in hole 1050-005:
  - including 47.94 g/t over 0.75;
  - including 89.3 g/t over 0.35 metres (as reported on November 16, 2016 press release).
- 8.37 g/t gold over 1.25 metres in hole 1040-002 (together with historical assays, this forms part of an intersection of 20.17 g/t gold over 6.21 m).

These results were not included in the 2016 MRE. Core from 134 drill holes has been reviewed and assay results for three batches totaling 275 samples have been received with results reported in this press release representing the third batch of 99 samples.

10 DRILLING

The issuer did not carry out any drilling on the Fenelon Mine Property.

All drilling programs completed to date on the Property have employed diamond drilling rigs with either BQ or NQ caliber core. The majority of the work has been completed during the winter months when the northern portion of the property is more readily accessible.

The first reported drilling program on the Property was conducted in 1993. In all, 351 holes (58,756 metres) have been drilled on the Property since then (Table 10.1).

In 1993, one hole for 185 metres was drilled on the area later named the Discovery Zone/Fenelon deposit.

In 1994, an 8-hole program totalling 1,426 metres was carried out in the same area.

In 1995, 69 new holes were added to the Property for a total of 17,400 metres.

In 1996, 14 holes (4,327 metres) were drilled on the Property.

In 1997, another 51 holes (9,787 metres) were added.

In 1998, Fairstar completed a drill program of six short holes totalling 201 metres to test the up-dip extension of the Discovery Zone.

In 2000, Taurus completed a 24-hole program totaling 992 metres on the Discovery Zone.

In 2002, a diamond drilling program was undertaken in the vicinity of the pit. Taurus drilled a total of 42 NQ-caliber holes for 2,351 metres. The holes were bored from the surface or from the bench built around the stripped area. All collars were surveyed. Acid tests were performed at 30-metre intervals to follow the deviation of holes. All casings were pulled. The aim of this program was to gain a better understanding of the mineralized zones, structures and locations. Holes drilled in 2002 targeted the known mineralized zones at a depth not exceeding 50 vertical metres.

In 2004, 62 holes, for a total length of 4,054 metres were drilled from underground. From these, 54 were NQ-caliber and located on 5–10 m drill spacing grids in the northern access drift on level 5213. The remaining eight (8) were located on production drifts. These holes were drilled to better define and determine the continuity of the mineralized zones.

In 2005, 12 holes (3,582 metres) were drilled in the vicinity of the Discovery Zone.

In 2006, an additional 27 holes (7,640 metres) were added in the vicinity of the Discovery Zone.

Finally, in 2011, an additional 35 holes (6,811 metres) were added to the Property.

After reviewing the drill data, InnovExplo is of the opinion that industry standards and best practices were employed during each program, although there is very little to no information on the quality control procedures, recoveries and handling procedures in the majority of the published reports.

A summary of diamond drilling on the Fenelon Mine Property is shown in Table 10.1. Figure 10.1 shows the drill hole locations on the Property, and Figure 10.2 shows a typical cross section.

Table 10.1 – Summary of diamond drilling exploration work on the Fenelon Mine Property

Year	DDH Count	Length (m)	Collar Location
1993	1	185	Surface
1994	8	1 426	Surface
1995	69	17 400	Surface
1996	14	4 327	Surface
1997	51	9 787	Surface
1998	6	201	Surface
2000	24	992	Surface
2002	42	2 351	Surface
2004	62	4 054	Underground
2005	12	3 582	Surface
2006	27	7 640	Surface
2011	35	6 811	Surface
Total	351	58 756	

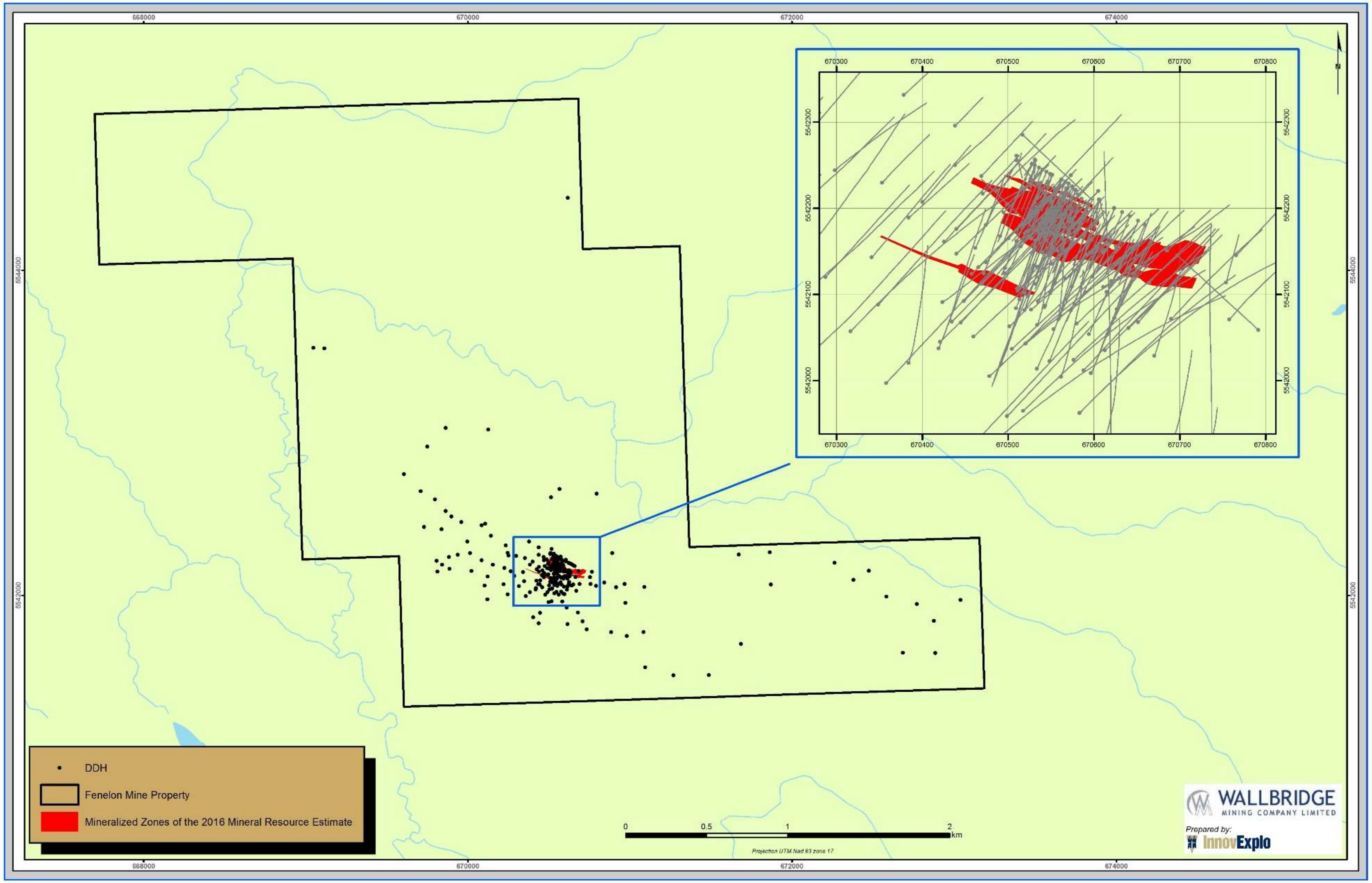


Figure 10.1 – Distribution of drill holes on the Fenelon Mine Property in relation to the mineralized zones involved in the current resource estimate

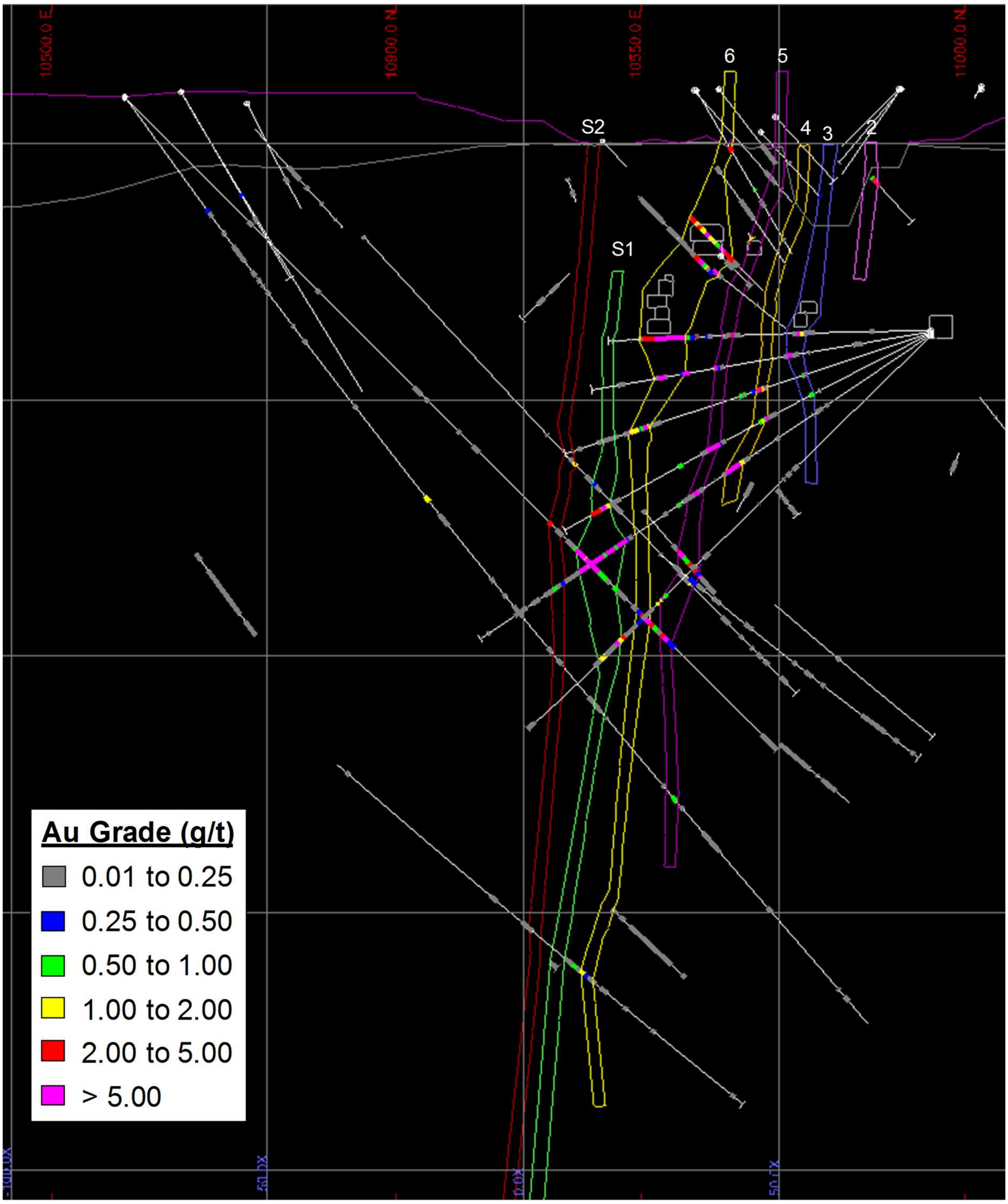


Figure 10.2 – Typical cross-section showing drill holes and mineralized zones used for the current resource estimate.
Note that mineralized zones were clipped to the bedrock for the resource estimate.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

The issuer did not conduct any drilling or sampling on the Fenelon Mine Property.

The author of this item has summarized the sample preparation, security and analytical methodology and protocols provided in past reports. In some cases, where the historical work was conducted before the implementation of NI 43-101 and ISO certification, the methodology and protocols were not reported or would not meet current standards. Data verification was typically via check assay procedures at a second analytical facility.

11.1 Recent Period

As of 2002, all core was sawed in half and the samples sent to Intertek Testing Services–Chimitec Laboratory in Val-d'Or for analysis. Some samples from the 2002 campaign that displayed typical alteration and mineralization styles were kept as office reference samples. These were sent to Montreal in core boxes, to be used for exhibits.

The core is stored at the Fenelon site. Some core drilled before 2000, which was stored at a farm near Rouyn-Noranda, was relogged by InnovExplo in 2005 and then brought to the Fenelon site in 2008. These are now wrapped on pallets.

The geologists who were managing and supervising the drilling programs sampled all mineralized sections. Very limited sampling has been undertaken systematically of the wallrock between individual mineralized zones in the Discovery Zone area. The core was sawed in half using a circular diamond saw: one half was sent to ALS Chemex in Val-d'Or for analysis and the other half kept for future reference. For the 2007 and 2008 campaigns, the core boxes were systematically photographed before being sawed.

Typically, samples were a standard length of 50 centimetres to a maximum length of 1.50 metre, generally reflecting the context of the mineralization. In some cases, the interval was chosen according to geological contacts, alteration styles or the presence of veining. The samples were carefully measured. The half-core remaining on site was placed back in the original core box and tagged to be easily re-identified. The samples to be analyzed were put in plastic bags with a water resistant numbered lab tag, and closed with a plastic tie wrap.

In 2004, during the underground drilling program, samples were sent to both Chimitec in Val-d'Or and ALS Chemex in Vancouver. Samples were prepped using primary crush to 90% minus 10 mesh, split for a 1000-gram sub-sample and pulverized to 90% minus 200 mesh. Standard fire assays were completed on a 50-gram pulp. QA/QC procedures consisted of check assaying at another laboratory. The data showed good correlation between the two laboratories, with a slight positive bias from ALS Chemex for samples with less than 300 ppb Au.

In 2005 and 2006, mineralized samples were analyzed by analytical package ME-ICP41 + Au23 and Au26 at ALS Chemex. In addition, whole rock samples were analyzed by the ME-XRF06 + Au23 + MEICP41 package at the same laboratory. The samples were collected from sawed half-core. Samples were individually bagged, sealed on site and transported to Val-d'Or for shipment to ALS Chemex. No specific quality control procedures are documented.

For the 2007 program, samples in the mineralized sections were analyzed at the ALS Chemex in Val-d'Or using the standard Au- AA23 and Au- AA26 packages for gold. For other metals, ALS Chemex used the ME-ICP41 package. During the program, some samples were chosen from specific sections for whole rock geochemistry and analyzed by the ME-XRF06 + Au23 + ME-ICP41 package.

In 2008, samples were analyzed at ALS Chemex in Val-d'Or according to the standard Au- AA23 and Au- AA26 packages for gold. For other metals, ALS Chemex used the ME-ICP41 package.

All core drilled since 2000 is stored at the Fenelon core storage site.

11.2 Early Period

During the early exploration programs from 1993 until 2000, the core was photographed, logged and split. Magnetic susceptibility and RQD measurements were also recorded. All potential mineralized zones were systematically sampled. The core splitter was carefully cleaned between each sample, and dismantled, cleaned and reassembled between each hole.

Several different laboratories were used on the property during this period and some of the historical reports contain little to no sampling data. Based on a review of all available reports, the author is not aware of any drilling or sampling factors that could have had a material impact on the accuracy and reliability of these results.

The samples from the 1993–1994 programs were analyzed by X-Ral Laboratories in Rouyn-Noranda. Samples were crushed to <10 mesh then 300 to 400 grams were pulverized to 90% <200 mesh. Gold analysis was by fire assay with atomic absorption finish. Assays returning results above 1 g/t Au were repeated with a gravimetric finish. Samples were also analyzed by nitric acid regia extraction with ICP finish for 32 elements. Check assaying was done by Swastika Labs in Swastika, Ontario. The Swastika results generally confirmed the X-Ral data and many results were higher than those from X-Ral. This was interpreted as a nugget effect. No security or preparation details were reported.

During the 1997 program, samples were sent to Techni-Lab for gold analysis by fire assay with atomic absorption finish. Samples returning higher than 1,000 ppb Au were systematically re-analyzed by fire assay with gravimetric finish. Samples with visible gold were sent for analysis using metallic sieve procedures. The rejects of 203 of these samples were sent to Chemex Labs in Vancouver. A good correlation was found between the two laboratories although Chemex had slightly higher values for samples containing less than 300 ppb Au. Some samples were also tested for Ag, Cu and Zn, although this stopped at some point. No security or preparation details were reported.

Also in 1997, samples from areas outside the Discovery Zone were analyzed for Au, Ag, Cu and Zn by Techni-Lab in Ste-Germaine-de-Boulé, Québec. Rejects for these samples were sent to Chemex Labs in Vancouver for comparison. A good correlation was observed, although the Chemex results show slightly higher values for samples with less than 300 ppb Au. No security or preparation details were reported.

The core from the Fenelon diamond drill programs carried out until 1998 is stored in a barn on a farm near the Rouyn-Noranda airport.

11.3 InnovExplo's Opinion

The author did not identify any significant analytical issues. InnovExplo is of the opinion that the sample preparation, analysis, QA/QC and security protocols used during the above mentioned drilling programs on the Fenelon Mine Property followed generally accepted industry standards, and that the data is valid and of sufficient quality to be used for mineral resource estimation purposes. Note that additional information on the QA/QC programs is provided in Item 12 – *Data Verification*.

12 DATA VERIFICATION

Wallbridge provided the diamond drill hole database that was used for the 2016 Fenelon Deposit Mineral Resource Estimate ("2016 MRE"; see Item 14). The discussion below refers only to holes drilled on the deposit and in the resource area, and does not apply to exploration holes that were drilled on the larger Wallbridge's Fenelon Property, far from the deposit, because those holes were not used for the resource estimate. The reviewed database is referred to as the "Fenelon Mine database" in this section.

The author, Catherine Jalbert, P.Geo., B.Sc., visited the Fenelon Mine Property on May 31 and June 1, 2016, accompanied by Alain Carrier, P.Geo., M.Sc., of InnovExplo, and Attila Pentek, P.Geo., of Wallbridge. During the site visit, the author was able to examine the logging facilities and the flooded open pit, review the core and drill hole collar locations, and resample eight (8) core samples and one (1) ore pad sample. Some of the data verification also took place before and after the site visit.

12.1 Wallbridge Mining Drilling

At the moment of the May 2016 site visit, by Catherine Jalbert, Wallbridge was in the process of acquiring the Fenelon Mine Property, no drilling was in progress during the site visit.

12.2 Historical Work

The historical work discussed in this report was validated by InnovExplo for the 2004 mineral resource estimate. The 2006 and 2011 drill holes were validated for the 2016 MRE.

12.3 Fenelon Mine Database

Two databases were sent to InnovExplo: one in GEMS format and the other in Geotic format. The databases were compared. The Geotic database contained seven (7) more holes, which were then added to the GEMS database even though they were not in the resource area. A total of 331 holes were selected (surface and underground) for the 2016 MRE. Of those, a subset of 230 holes cut across the mineralized zones. Multiple channel and muck samples were also incorporated into the GEMS database, but these data were not validated as they were not used for interpolation.

12.3.1 Coordinate System

The decision was made to work in local coordinates. All 3D objects were in local coordinates in the GEMS Project, as well as drill hole positions. The conversion formula, from local to UTM NAD 83 Zone 17, was calculated by the surveying firm J L Corriveau & Assoc. Inc.

12.3.2 Drill Hole Locations

All surface drill holes on the Fenelon Mine Property have been surveyed either professionally or by a handheld GPS unit. Nine (9) holes from the 2011 Balmoral drilling program were visited, and good accuracy was obtained between the GEMS database coordinates and the on-site reading from a handheld GPS unit (Fig. 12.1).

InnovExplo concluded that the collar locations for the 268 surface drill holes are adequate and reliable. See Table 12.1 for the comparison.

Ninety-five (95) holes were assigned a new elevation based on a 2011 professional survey on older and recent drill holes.

Figure 12.1 – Examples of on-site collar location verification (FAB-11-12 and FAB-11-26)

Table 12.1 – Coordinate comparison between the database and onsite measurements.

Collar	Field Measurements (UTM)		Field Measurements (Local)		Database		Differences (m)	
	Easting	Northing	Easting	Northing	Easting	Northing	Easting	Northing
FAB-11-05	670481	5541992	10477.06	10760.34	10477.00	10761.99	0.06	-1.65
FAB-11-10	670510	5542085	10509.20	10852.18	10507.99	10851.42	1.21	0.76
FAB-11-11	670510	5542085	10509.20	10852.18	10507.99	10851.42	1.21	0.76
FAB-11-12	670511	5542104	10510.49	10871.09	10510.58	10870.35	-0.09	0.74
FAB-11-14	670503	5542147	10504.26	10914.14	10503.97	10914.58	0.29	-0.44
FAB-11-15	670503	5542147	10504.26	10914.14	10503.97	10914.58	0.29	-0.44
FAB-11-17	670568	5542090	10566.50	10855.64	10566.21	10857.59	0.29	-1.95
FAB-11-26	670606	5542082	10604.62	10846.61	10603.78	10843.40	0.84	3.21
FAB-11-29	670625	5542075	10622.98	10839.13	10623.68	10839.78	-0.70	-0.65

12.3.3 Downhole Survey

Downhole surveys were available for all the holes used for the 2016 MRE. The most recent drill holes had Flexit multi-shots taken every 3 metres. For pre-2006 drilling, the testing was mostly acid and Pajari, generally at every 30 metres. All information was mathematically reviewed for all drill holes in the database to identify anomalies, and visual checks were performed on 100% of the downhole surveys. No modifications were made to the database and it was considered valid and reliable.

12.3.4 Assays

InnovExplo was granted access to the certificates of assays for the latest drilling campaign that took place in 2011. The 2006 certificates were already in InnovExplo's possession since the program had been executed and supervised by a team from InnovExplo.

Minor errors of the type normally encountered in a project database were identified and corrected. The final database is considered to be of good overall quality. InnovExplo considers the Fenelon Mine database to be valid and reliable.

Some inconsistencies were observed in the reported average gold grades. A new average was then calculated according to the following order of priority:

- Metallic sieve results (mean value if multiple);
- Gravimetric results (mean value if multiple);
- Fire assays results (mean value if multiple).

The new average was incorporated into the database and was used for the 2016 MRE.

12.3.5 QA/QC

The Wallbridge mining team has not established a QA/QC protocol because they have not carried out any drilling.

However, QA/QC data from previous drilling programs were available in the Geotic database and these were validated. A total of 507 samples were listed, divided into twelve (12) types (Table 12.2). A few minor issues were noticed, but overall, the QA/QC protocol was considered valid (Figs. 12.2 and 12.3).

Table 12.2 – Types of QA/QC listed

Type	# of samples
STD 14A	35
STD 1H	2
STD 1P5C	5
STD 2F	19
STD 2G	20
STD 30B	35
STD 5F	40
STD 8A	90
Ciment	9
STD OxK35	3
STD OxL34	5
Quartz (blank)	244
Total	**507**

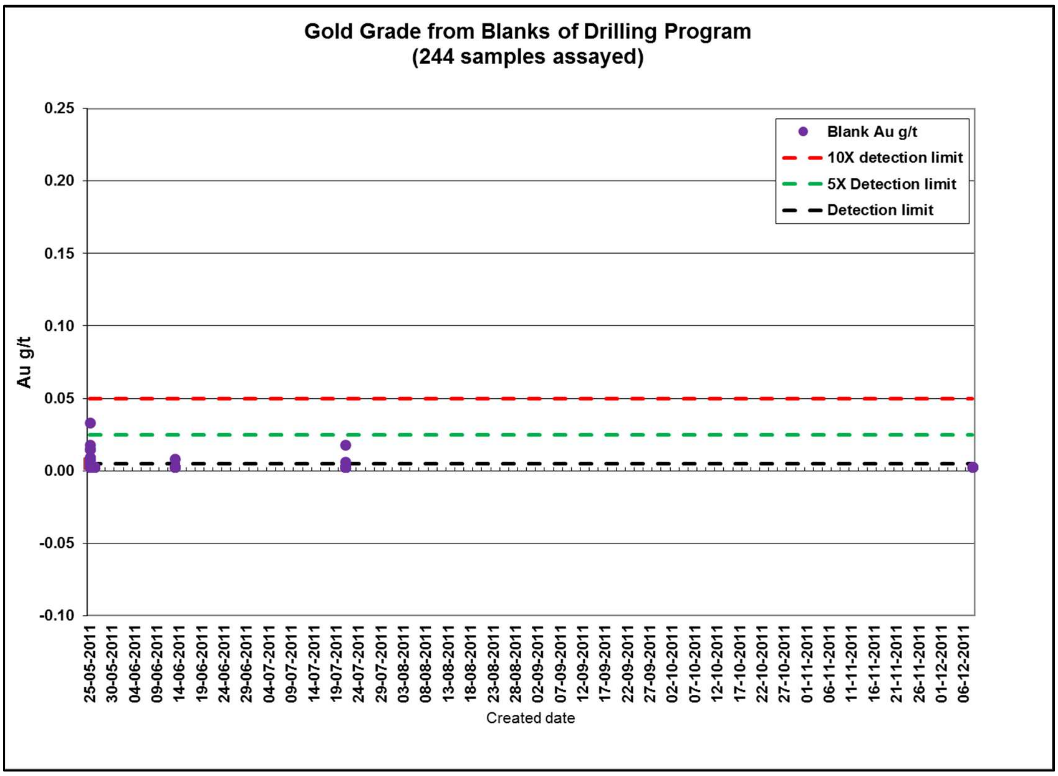


Figure 12.2 – Blank results from the Geotic database

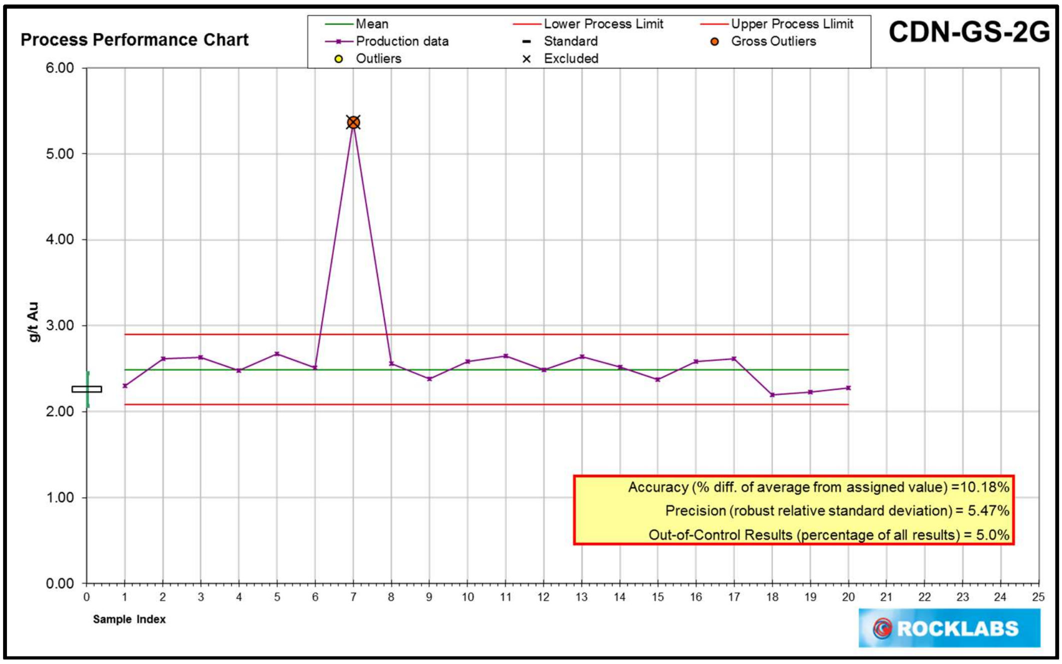


Figure 12.3 – Standard CDN-GS-2G results from Geotic database
(one incoherent sample was considered as a mixed type)

12.3.6 Voids

Wallbridge provided InnovExplo with data on underground voids. Most of the voids were already available in 3D, modelled by previous owners, but some stope contours were modelled by the Wallbridge team. Those voids had never been converted into 3D format due to the abrupt closures of the mine in the mid-2000s. They were modelled using the data from underground mapping. Based on the available data, the voids (drifts and stopes) in the GEMS project are considered accurate.

12.4 Independent sampling

The author reviewed multiple mineralized drill hole intersections and resampled eight (8) core samples from three (3) different drill holes using the quarter-split method. One (1) other sample was taken from the ore pad.

All core boxes were labelled and properly stored outside, either under roofed racks or cross-spaded on the ground (Fig. 12.4). Sample tags were still stapled to boxes, which facilitated the validation of mineralized intervals (Fig. 12.5).

Low-grade samples yielded results that are consistent with the original results (Table 12.3). For higher-grade samples, the results varied considerably, but this is likely due to the high nugget effect, which is commonly encountered in this type of deposit.

Figure 12.4 – Core storage on the Fenelon Mine Property

Figure 12.5 – Example of re-sampling in hole FAB-11-20A

Table 12.3 – InnovExplo's re-sampling results

Original Data					Re-sampled Data					
Hole-ID	From	To	Sample-ID Original	Au ppm (original)	Sample-ID	Weigth (kg)	Au ppm (AU-AA26)	Au ppm (AU-GRA22)	Specific gravity (rock)	Specific gravity (pulp)
FA-06-297	120	121.1	45222	1.04	P227201	1.22	3.93	3.27	2.8	2.82
FA-06-297	121.1	122.2	45223	21.7	P227202	1.14	12.2	12	2.64	2.81
FA-06-297	122.2	123.2	45224	0.04	P227203	1.04	0.02		2.75	2.85
FAB-11-33	75.06	75.5	K440222	2.97	P227204	0.43	3.88	3.46	2.54	2.76
FAB-11-33	75.5	76.35	K440223	4.19	P227205	0.8	2.91		2.6	2.73
FAB-11-33	76.35	77	K440224	0.102	P227206	0.68	0.12		2.69	2.81
FAB-11-20A	204	205	K439092	0.028	P227207	1.12	0.02		2.73	2.89
FAB-11-20A	205	206	K439093	3.07	P227208	1.03	7.37	7.49	2.72	2.89
Muck					P227209	1.49	>100	177	2.69	2.74

12.5 Conclusion

Overall, the author is of the opinion that the data validation process, from site visit to database, demonstrates the validity of the project on the Fenelon Mine Property. The database is of sufficient quality to be used for a resource estimate.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

The laboratory testwork described below comprises tests done in 1997 at Centre de Recherche Minérales of Ste-Foy, Quebec ("CRM"; now COREM). Most of the gold recovery results were obtained during the processing of bulk samples at the custom processing facility of the Camflo Mill in Malartic. The first bulk sample was collected in 2001. It was taken during open pit mining of zones 102, 103 and 104, and was processed as two distinct batches. The second bulk sample was collected in 2004 from an underground ramp; most of the sample came from zones 105 and 106. In all cases, the reported gold recoveries were good despite some reported losses experienced during processing. Due to the presence of sulphides (pyrrhotite and chalcopyrite), some additional laboratory tests were proposed and completed. The results confirmed the gold recovery for the deeper part of the zones as well as the untested zones (107 and S1), which represent nearly 31% of the resources. All the relevant metallurgical testwork information and bulk sample data are included in Appendix III.

13.1 Preliminary Characterization at CRM

A pre-feasibility study was completed in 1997 by CHIM International ("CHIM") of Montreal, Quebec. The mandate for the preliminary characterization of the Fenelon gold mineralization was given to CRM. The testwork was completed in October 9, 1997 on a reportedly representative 20-kg composite core sample submitted by CHIM. However, no description of the sample origin or location is available to confirm that it was indeed representative. The average calculated gold recovery was 99.1%, suggesting that Fenelon mineralization responds extremely well to the conventional cyanidation process. The grade of the sample was more than twice the suggested mining grade. The following summary is extracted from the CHIM and CRM reports.

The 20-kg sample was crushed to -10 mesh and mixed, then three cuts were assayed by metallic screen to determine the head assay.

Table 13.1 – Sample grade assays

Sample No.	1	2	3	Average
Assays (g/t)	24.4	20.7	26.2	23.8

Subsamples weighing between 1,000 and 3,000 g were prepared for further testing. The calculated heads grade for those subsamples were 23.7 g/t Au for the flotation test sample, and 23.5 and 26.2 g/t Au for the two cyanidation tests, which are in line with the metallic screen assays. The composite ore sample was also analyzed for multiple elements. These results do not reveal the presence of any deleterious elements for ore processing except for copper (0.24%), which could increase cyanide consumption. Graphitic carbon was 0.04% C, but does not appear to be preg-robbing. Specific gravity was measured as 2.82 g/cm^3.

Table 13.2 – Multi-element analysis

Éléments	
SiO2 (%)	57.6
Al2O3 (%)	11.2
Fe2O3 (%)	7.96
MgO (%)	5.46
CaO (%)	5.75
Na2O (%)	2.36
K2O (%)	1.44
TiO2 (%)	0.45
MnO (%)	0.11
P2O5 (%)	0.08
PAF (%)	3.59
Ag (ppm)	8.4
As (ppm)	290
C graphitique (%)	0.04
Cu (ppm)	2400
Pb (ppm)	35
Hg (ppb)	60
S (%)	2.02
Sb (ppm)	<5
Zn (ppm)	130

Densité (g/cm³)	2.8235

Teneur en or (g/t)	
Métallique 1	24.4
Métallique 2	20.7
Métallique 3	26.2
Souche calc. cyan 1	23.5
Souche calc. cyan 2	26.2
Souche calc. flott.	23.7
Moyenne	24.1
Écart-type	2.0

A mineralogical study on the flotation concentrate revealed that gold particles are free and generally fine grained. In fact, 85% of the 15 gold particles observed had grain sizes between 50 and 80 microns.

A gravimetric separation test in heavy liquid (3.3 g/cm^3) returned a gold recovery of 57.9% at a grind of 48.3% -200 mesh, suggesting that at this grind size, a large proportion of the gold is not free. The grades of the concentrate and tail were 462.9 and 10.6 g/t Au, respectively. Gravimetric separation may improve coarse gold recovery and minimize uncontrolled coarse gold losses and buildup in the circuit.

Two bottle cyanidation tests were conducted at 85% -200 mesh for 45.5 hours. The cyanide consumption was found to be quite high at 2.3 kg/t at a cyanide concentration of 1 g/t. This may have been caused by copper dissolution during the extensive leach time, but the pregnant solution was not analyzed for copper. Lime consumption was 1.9 kg/t. The average grade of the leach tail was 0.23 g/t Au. The average gold recovery was 99.1%, suggesting that Fenelon mineralization responds extremely well to the cyanidation process. However, the leach time (45.5 hrs) and cyanide consumption were considered above normal, possibly caused by elevated copper and pyrrhotite concentrations in the mineralization. The leach time may also be due to the presence of coarse gold in the ore. The report recommended further studies to determine whether these problems could be mitigated, although no additional testwork was performed.

The flotation test was conducted at a grind size of 85% -200 mesh (74 µ) and returned a calculated gold recovery of 96.5%, which is also excellent. The weight of the concentrate was approximately 16% of the feed. The grade of the concentrate and reject were 329 and 1 g/t Au, respectively. Further studies were recommended to test whether the concentrate could contain 22 to 24% copper and still be amenable for smelting. It was also recommended that flotation be examined as a potentially economically attractive treatment process for Fenelon ore.

The Bond Work Index was calculated at 10.5 kwh/t, indicating that Fenelon mineralization falls in the category of "soft ore" and can be easily ground by a standard rod mill–ball mill combination.

The CHIM study also evaluated various custom milling options that were available when the study was prepared. However, no information is available on the results of this review.

In addition, CRM tested the acid generation potential (AGP) of the submitted sample using the BC Research method. It was determined that the cyanidation reject had a negative AGP. The results yielded calculated AGPs of -28.2 and -32.4 kg/t H_2SO_4 for cyanidation tests 1 and 2, respectively. These results demonstrate the cyanidation tail was not acid generating.

13.2 Bulk Sample Processing at the Camflo Mill

13.2.1 First bulk sample

In 2001, a total of 13,713 dry metric tons of ore from the bulk sampling program was hauled by truck for approximately 300 km to the Camflo Mill operated by Richmont Mines Inc. The mill processed the bulk sample as two separate batches. The first batch of 5,187 tonnes was processed between May 30 and June 4, 2001. The second batch of 8,526 tonnes was processed between June 27 and July 06. The calculated average head grade was 9.84 g/t Au and 3.00 g/t Ag, with a calculated gold recovery of 97.1%.

Table 13.3 – Summary of milling results, 2001

Process	Direct Cyanidation, Merrill Crowe
Tonnage rate	43 t/h (metric tons per hour)
Net power draw	10.5 kWh/t (kilowatt-hours per metric ton)
Final grind	± 85% -200 mesh and ± 65% -325 mesh
Head grade	9.84 g/t Au
	3.00 g/t Ag
Gold recovery	97.1%

Table 13.4 – Comparison of leaching time versus % gold dissolution

Time	Gold dissolution/leach	Total gold dissolution
Grinding		55.4%
8 Hours	5.0%	59.4%
16 Hours:	17.5%	72.9%
24 Hours:	31.5%	86.9%
32 Hours:	36.0%	91.4%
40 Hours:	40.3%	95.7%
45 Hours:	42.4%	97.8%

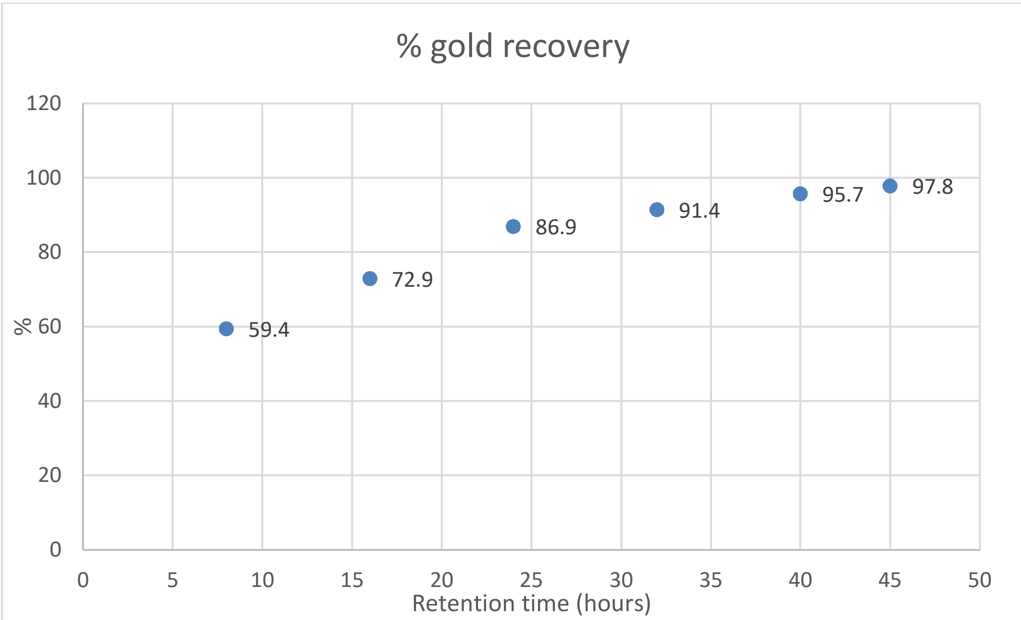


Figure 13.1 – Kinetic gold dissolution

Table 13.5 – Summary of reagent consumption during processing of the Fenelon ore.

Reagent	Consumption (kg/t)
Sodium cyanide	1.00
Quick lime	1.36
Lead nitrate	0.13
Lead acetate	0.001
Zinc dust	0.022
Flocculent	0.015
Anti-scaling	0.025

The processing of the Fenelon ore at Camflo matched the results obtained by the preliminary testing done at CRM. However, cyanide consumption was significantly lower at 1.0 kg/t compared to 2.3 kg/t in the CRM test results. It may be the result or effect of ore dilution during the mining process. The head ore grade of 9.89 g/t Au represents approximately half the mine's expected grade. The conclusion of this bulk sample was that an important dilution factor occurred during open pit mining.

Only minor issues were encountered during ore processing. Higher than expected soluble tailings gold losses occurred while processing the first ore batch due to premature high pressure in gold presses. This problem was reported and corrected by injection of an anti-scaling compound into the clarifier tank while processing of the second batch. In the refining process, gold buttons were poured prior to the bars to separate the mat from the gold. It was suggested that this problem could be corrected by an appropriate choice of flux and/or by controlling the redox by increasing the bleeding of the barren solution during the process.

The Camflo milling facilities were adequate for processing Fenelon ore. Fenelon ore has a relative low work index and contains some chalcopyrite. Finally, Jolicoeur (2001) recommended that a SAG mill and CIL processing should be taken in consideration as an alternative for a new milling facility.

13.2.2 Second bulk sample

In 2004, financial difficulties encountered by the former owner and the rapid closing of the mining operations resulted in a lack of adequate information for the second bulk sample. However, according to the January 2005 report by InnovExplo, the second bulk sample sent by the mine to the Camflo Mill had a tonnage of 7,757 metric tons grading 9.01 g/t Au. The tonnage and grade provided by the mill and reported in the report by E. St-Jean of Laboratoire LTM Inc. was 8,169.20 metric tons at a calculated head ore grade of 10.25 g/t Au. St-Jean was an expert who supervised the ore treatment at the Camflo mill on behalf of the previous owner. The difference in tonnage sent by the mine and that received by the mill is 5% and represents an acceptable difference.

The report does not take into account the amount of gold in the matte and slag and what was recovered after cleaning the tank house because they were not analyzed. It is probable that they contain several ounces of gold (from 5 to 10 oz).

It was reported that a mill malfunction occurred again during the processing of the second bulk sample. The report said the problem occurred on September 11 as pressure in the presses increased abnormally. In the evening, they blew the presses by insufflating pressurized air into them. The colour test showed signs of gold loss over a period of six hours during the night, but that the situation had gone back to normal. The quantity of gold lost to the waste during the mill malfunction resulted in the loss of about 90 ounces of gold, which would normally be recoverable.

For the total of 8,169 metric tons, the mill feed grade was estimated at 10.25 g/t, with a recovery of 95.5%. After the final inventory of the mill and adjustment, the grade was calculated at 10.70 g/t for a tonnage of 8,073 metric tons and a gold recovery of 93.5%. If the 90 ounces lost to the mill malfunction is included in the mill reconciliation or accounted for in the recovery calculation, the total gold recovery may be close to 97%.

Table 13.6 – Summary of milling results per operating days in 2004 (daily mill report)

Date	Dry tonnes	Feed grade	Gold recovery
	tonnes	g/t	%
8/09/2004	497	10.11	98.51
9/09/2004	1083	14.26	98.25
10/09/2004	1016	12.21	97.67
11/09/2004	1076	12.96	94.27
12/09/2004	1081	12.45	91.49
13A/09/2004	664	10.63	93.48
13B/09/2004	347	12.17	94.20
High grade	**5764**	**12.41**	**95.37**
14/09/2004	924	6.45	94.19
15/09/2004	1008	3.77	96.61
16/09/2004	474	5.11	97.31
Low grade	**2406**	**5.06**	**95.82**
Total	**8169**	**10.25**	**95.50**

Table 13.7 – Summary of milling results 2014 (after final inventory and adjustment)

Process:	Direct Cyanidation in Merrill Crowe
Tonnage rate:	45.3 t/h (metric tons per hour)
Net power draw:	11.52 kWh/t (kilowatt-hours per metric ton)
Work index:	8.85 kWh/t
Final grind:	± 80% -200 mesh
Head grade:	10.70 g/t Au (grams per metric ton)
Gold recovery	93.5%

After processing this second bulk sample, it was recommended to examine the copper flotation alternative again and take a look at the potential of graphitic material occurrence in the ore and make sure it will not affect the gold recovery. Finally, it was recommended to increase the amount of lead addition to the gold recovery press to prevent the recurrence of the high pressure and gold losses.

13.3 Additional laboratory testwork, 2016

Based on the results of the two bulk samples discussed above, the size of the deposit and the amount of available ore, it appears that good gold recovery can be expected by conventional leaching. However, due to the events that occurred during the previous testwork, it was decided that new confirmation tests should be performed on material from the deeper horizon and a zone that has not yet been mined (Zone 107). Nine (9) leaching tests were conducted under similar conditions to the 1997 testwork. The intent was to confirm the gold recovery achieved to date.

13.3.1 Sample details

The sources of the sample and known test details are as follows:

- Zone 107 or S1 (31% resource): 3 leach tests (no data available).
- Zone 106 (31% resource): 3 leach tests (process difficulty & gold losses reported).
- Zone 105 (15% resource): 2 leach tests (recovery confirmation).
- Zone 104 (11% resource): 1 leach test (recovery confirmation).

The sample location and sample list are included in Appendix III.

Table 13.8 – Sample assays

Sample number	Zone	Composite Weight	Head Assays
		Au (g/t)	Au (g/t)
1130-001	107 (S1)	2827	28.9
1050-007	107 (S1)	2368	26.3
FAB-11-25	107 (S1)	2757	6.8
1050-008	106	2246	6.6
1040-002	105	3288	6.2
1045-001	106	2649	24.5
FAB-11-12	105	3375	9.1
1040-001	106	2901	35.1
1040-005	104	2400	53.7

The leaching tests were conducted under similar conditions to the test done in 1997. Because of the copper occurrence additional assays were done to determine copper grade and track other elements in the ore and leached solutions.

13.3.2 Leaching conditions

- 48-hour tests, 2 L reactor, 1 kg ore sample (50% solids) at room temperature;
- pH of 11.5, adjusted with lime;
- NaCN concentration of 1 kg/t;
- pH and NaCN concentrations maintained;
- Au in solution measured at 24 and 48 hours;
- Au in solids by fire assay at 48 hours;
- ICP-MS on solid and solution from the leached tailings.

13.3.3 Testwork results

The primary results returned mix results, lower than anticipated. A review of the results showed a high variation in the gold assays in the leaching tail, which normally indicates the presence of coarse gold or incomplete leaching of gold. The leaching kinetic also appears slower than tests done in the past. The ICP results do not reveal any problems with copper or any other element. No correlation can be made with copper dissolution. No oxygen or cyanide problems were reported during the test, either. The unexpected results may therefore be due to the quality of the samples, which were collected from old diamond drill holes (10-12 years old). Considering these facts and the time schedule for the study, it was decided to proceed with an intensive leaching test on the leaching residue to quantify the amount of residual gold not already leached. During the initial test, the target grind was not achieved over most of the test. The intensive leach tests were done on the reground leaching tail with the intent of working with material closer to the target grind size. Two additional tests have been done without the reground tail on the two most problematic samples in order to verify the effect of the grind.

The ICP assays provided a copper grade between 0.05% and 0.34% Cu, and the cyanide consumption is directly proportional to the copper grade. The copper concentration in the pregnant solution ranges from 40 ppm to 480 ppm.

Table 13.9 – Leaching results

NaCN data										
	Time (h)	**1130-001**	**1050-007**	**FAB-11-25**	**1050-008**	**1040-002**	**1045-001**	**FAB-11-12**	**1040-001**	**1040-005**
[NaCN] (mg/L)	1	447	720	713	757	672	766	762	642	717
	3	729	931	853	919	891	818	901	963	844
	6	838	793	943	925	969	950	725	1009	941
	24	632	1077	736	864	888	855	1183	809	918
	48	530	1011	797	999	1049	881	1270	882	851
NaCN (g) Add.	1	0,558	0,292	0,287	0,245	0,329	0,236	0,239	0,359	0,291
	3	0,274	0,000	0,149	0,000	0,111	0,186	0,100	0,000	0,158
	6	0,167	0,211	0,000	0,000	0,00	0,000	0,283	0,000	0,000
	24	0,369	0,000	0,269	0,144	0,118	0,148	0,000	0,199	0,000
NaCN consumption(g/t)	1	553	280	287	243	328	234	238	358	283
	3	826	363	441	321	433	418	341	391	444
	6	994	501	506	315	466	475	619	345	505
	24	1371	433	713	376	547	570	456	545	528
	48	1860	499	943	389	506	700	369	676	595
Au leaching results										
Au leached (mg/L)	24	11,7	6,8	5,6	2,6	3,6	14,0	4,5	14,8	36,7
	48	16,2	11,0	9,8	5,3	7,6	20,9	8,9	20,8	43,7
Au solid (g/t)	48	1,96	7,31	0,725	0,17	0,91	0,40	0,30	14,3	13,6
Au Feed (g/t)	(-)	28,9	26,3	6,8	6,6	6,2	24,5	9,1	35,4	53,7
Au Feed recalculed (g/t)	(-)	18,2	18,3	10,5	5,5	8,5	21,3	9,2	35,1	57,3
Au recovery (%)	48	89,2	60,1	93,1	96,9	89,3	98,1	96,8	59,3	76,3
Copper assays										
Cu (liquid, mg/L)	48	482	53	236	55	43	156	40	141	108
Cu (Solid, mg/kg)	48	2940	673	3000	522	377	1860	579	2230	1370
Cu Feed recalculed (%)	(-)	0,342	0,073	0,324	0,058	0,042	0,202	0,062	0,237	0,148
Lime consumption (kg/t)	48	1,79	1,87	1,83	* 0,98	*1,05	1,81	2,02	*0,93	2,03
					*Lime consumption are underestimated related to p»H senser problem.					

Detail on pyroanalyses of leaching solid tail

	Au (g/t)								
Analysis	**1130-001**	**1050-007**	**FAB-11-25**	**1050-008**	**1040-002**	**1045-001**	**FAB-11-12**	**1040-001**	**1040-005**
1	0,14	4,32	1,09	0,25	1,7	0,65	0,27	14,6	12,9
2	3,77	10,3	0,36	0,09	0,11	0,14	0,32	14	14,3
Average	1,96	7,31	0,725	0,17	0,91	0,40	0,30	14,3	13,6

Table 13.10 – ICP results

Scan ICP, Leaching solution

Elment	1130-001	1050-007	FAB-11-25	1050-008	1040-002	1045-001	FAB-11-12	1040-001	1040-005
Be	<0,02	<0,03	<0,02	<0,03	<0,03	<0,02	<0,03	<0,03	<0,03
Na	1100	671	850	553	648	738	666	674	592
Mg	0,02	0,05	0,02	0,29	0,32	0,04	0,1	0,36	0,03
Al	7,92	2,45	6,51	0,38	0,91	3,97	0,98	2,22	2,19
Si	3,1	6,6	2,7	10,2	7,5	4,6	11	5,4	3,7
P	<0,6	<0,8	<0,6	<0,8	<0,8	<0,6	<0,8	<0,8	<0,8
K	23,6	12,9	11,7	29,2	21,8	21,2	27,6	10,5	22,9
Ca	59,1	35,3	55,8	7,7	3,7	28,1	29,7	1,6	34,7
Sc	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002
Ti	<0,004	<0,005	<0,004	<0,005	<0,005	<0,004	<0,005	<0,005	<0,005
V	0,002	0,008	<0,002	0,05	0,049	0,007	0,029	0,017	0,01
Cr	<0,002	<0,003	<0,002	<0,003	<0,003	<0,002	<0,003	<0,003	<0,003
Mn	<0,002	<0,003	<0,002	0,004	<0,003	<0,002	<0,003	0,004	<0,003
Fe	3,6	3,55	4,03	2,91	5,83	4,88	3,03	8,84	4,16
Co	0,195	0,104	0,122	0,113	0,318	0,094	0,18	0,156	0,192
Ni	0,566	0,673	0,844	1,06	4,39	0,685	0,741	0,741	1,19
Cu	482	52,5	236	55,3	42,5	156	39,6	141	108
Zn	2,91	7,04	2,92	1,92	3,42	3,15	1,72	1,82	1,68
Ga	0,008	<0,005	0,005	<0,005	<0,005	0,006	<0,005	0,006	<0,005
Ge	<0,6	<0,8	<0,6	<0,8	<0,8	<0,6	<0,8	<0,8	<0,8
As	0,019	0,01	0,007	0,009	0,014	0,005	0,021	0,068	0,009
Se	<0,4	<0,5	<0,4	<0,5	<0,5	<0,4	<0,5	<0,5	<0,5
Sr	0,31	0,232	0,413	0,113	0,0642	0,195	0,344	0,0207	0,282
Zr	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002
Mo	0,12	<0,05	0,022	<0,05	<0,05	0,02	<0,05	<0,05	<0,05
Ag	8,75	3,42	4,35	1,91	2,17	7,82	4,4	7,79	9,58
Cd	0,003	0,013	0,002	0,003	0,006	0,005	0,003	0,003	0,004
Sb	0,021	<0,05	0,011	<0,05	<0,05	0,007	0,058	<0,05	<0,05
Ba	0,0031	0,003	0,002	0,0034	0,0019	0,0035	0,0073	0,0016	0,0057
Tl	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2
Pb	<0,002	0,004	<0,002	<0,003	<0,003	<0,002	<0,003	<0,003	<0,003
Bi	<0,08	<0,1	<0,08	<0,1	<0,1	<0,08	<0,1	<0,1	<0,1
Th	<0,0004	<0,0005	<0,0004	<0,0005	<0,0005	<0,0004	<0,0005	<0,0005	<0,0005

Scan ICP, Leaching solid

Élément	1130-001	1050-007	FAB-11-25	1050-008	1040-002	1045-001	FAB-11-12	1040-001	1040-005
Be	<1	<1	<1	<1	<1	<1	1	<1	1
Na	7530	27400	18500	18800	6340	38500	12100	27900	25000
Mg	6250	26500	30900	60000	65000	17300	63100	18700	45000
Al	80500	63900	62200	58300	53900	67300	50400	59500	56000
K	39000	13000	12300	8130	6150	8830	16200	7980	11100
Ca	17000	36700	28000	62000	66700	18400	64600	29500	52900
Sc	10,9	12,7	14,1	20,2	21,6	8,7	21	8,9	15,5
Ti	1700	1120	733	2710	3400	1800	2770	1090	2930
V	71,3	115	116	145	189	75	126	69,5	135
Cr	168	360	557	1150	1370	300	1050	367	781
Mn	211	724	566	1200	1140	412	1130	796	923
Fe	35900	52300	66800	59400	67600	39600	59800	37400	68700
Co	47,8	37,5	45,1	48,6	57,9	29	48,4	26,5	60
Ni	134	161	248	396	446	158	379	148	327
Cu	2940	673	3000	522	377	1860	579	2230	1370
Zn	167	503	151	87	349	147	73	204	95
As	82	25	48	<5	34	10	56	25	37
Se	<20	<20	<20	<20	<20	<20	<20	<20	<20
Sr	129	277	214	387	356	272	278	232	455
Mo	16,8	17,1	19,3	15,5	15,6	17,4	15,6	18	19,2
Ag	4,2	0,6	3,1	0,8	<0,5	1,3	1,1	1,9	1,4
Cd	1,4	1	0,8	<0,5	0,9	<0,5	<0,5	0,9	0,8
Sb	<10	<10	13	<10	<10	<10	<10	<10	<10
Ba	617	309	293	194	104	348	463	333	310
Tl	<20	<20	<20	<20	<20	<20	<20	<20	<20
Pb	33	170	25	12	23	76	79	20	19
Bi	84	7	109	7	<5	6	52	15	13
Th	5,6	1,6	2,6	2,9	1	2,6	3	2,4	2,9

13.3.4 Intensive leaching conditions of the leaching tail

Table 13.11 – Leach tail grind size and regrind conditions

Test	Sample No	P80 (µm) leach tail	Regrind time
1	1130-001	86,9	3 min
2	1050-007	74,9	3 min
3	FAB-11-25	53	1 min
4	1050-008	61,6	1 min
5	1040-002	71,6	3 min
6	1045-001	74,3	3 min
7	FAB-11-12	96,5	4 min
8	1040-001	89	4 min
9	1040-001	89	none
10	1040-005	103	4 min
11	1040-005	103	none

- Leaching time: 2h;
- NaCN concentration: 5%;
- Leachwell concentration: 2%;
- NaOH concentration: 0.7%.

13.3.5 Intensive leach testwork results

The intensive leach on the initial leach tail finalized the gold dissolution. The combined results provided similar results achieved in the 1997 testwork, and more in line with the bulk sample gold recovery. The final gold extraction reached 98% to 99.5% recovery. The testwork done with and without the regrind show similar results (plus 1% with regrind). It appears that the initial lower gold recovery in the initial test related more to sample quality (~10 year-old sample) or the presence of coarse gold than the grind size. Given the age of the available samples, the objective of this additional test was to verify gold leachability and not to quantify the gold recovery. The results also provide some information on copper and other metals present in the ore.

Table 13.12 – Leaching results

Sample number	Au liquid initial leach (mg/L)	Au liquid leach tail intensive leach (mg/L)	Au solid leach tail Intensive leach (mg/kg)	Feed calculated (mg/kg)	Gold recovery (%)
1130-001	16.2	2.5	0.38	19.1	98.0
1050-007	11	14.8	0.05*	25.9	99.8
FAB-11-25	9.81	0.59	0.17	10.6	98.4
1050-008	5.33	0.12	0.05*	5.5	99.1
1040-002	7.55	0.44	0.05*	8.0	99.4
1045-001	20.9	1.97	0.13	23.0	99.4
FAB-11-12	8.93	0.47	0.06	9.5	99.4
1040-001	20.8	13.4	0.15	34.4	99.6
1040-001 (Without regrind)	20.8	11.5	0.47	32.8	98.6
1040-005	43.7	19.8	0.36	63.9	99.4
1040-005 (Without regrind)	43.7	22.6	0.98	67.3	98.5

**0.05 mg/kg represent the limit of detection*

13.3.6 Gold recovery

A gold recovery of 97% seems achievable considering the historical and recent testwork results. Based on the historical testwork, the bulk sample results and the recent 2016 testwork, it appears that the deeper resources and Zone 107 (S1) can reach similar gold recoveries to those achieved in the past. However, it will be critical to control the liquid losses during ore processing or the final gold recovery will be affected, as happened during the 2004 bulk sample processing. Although high-pressure events occurred while processing both bulk samples, only the 2004 sample was negatively affected. Attention to the gold precipitation process conditions will be necessary, and proper control must be exercised to prevent copper precipitation. It may be appropriate to track the copper grade and optimize ore mixing in order to control the copper grade and sulphide variation in the mill feed. The amount of free gold suggests that gravity recovery may help control potential gold losses or trapping in the circuit during ore processing.

14 MINERAL RESOURCE ESTIMATES

The 2016 Fenelon Deposit Mineral Resource Estimate presented herein (the "2016 MRE") was prepared by Pierre-Luc Richard, P.Geo., and Catherine Jalbert, P.Geo., using all available information. The 2016 MRE was prepared as part of a mandate assigned by Wallbridge Mining Company Ltd ("Wallbridge") in 2016. It was originally presented in the report titled "Technical Report and Mineral Resource Estimate for the Fenelon Mine Property (according to National Instrument 43 101 and Form 43 101F1"), dated August 17, 2016. The Fenelon deposit has seen both underground and open pit operations in the past, and is also known as the Fenelon mine.

The 2016 resource area measures 500 metres along strike, 210 metres wide and 280 metres deep. The resource estimate is based on a compilation of historical and recent diamond drill holes and wireframed mineralized zones largely inspired by previous work and Wallbridge's interpretation. The final model was constructed by InnovExplo.

The mineral resources herein are not mineral reserves as they have no demonstrable economic viability. The result of this study is a single Mineral Resource Estimate for eight (8) mineralized zones (coded 102 to 109). The estimate includes Measured, Indicated and Inferred resources for an underground scenario. The effective date of the estimate is July 5, 2016, based on compilation status and cut-off grade parameters.

14.1 Drill Hole Database

The GEMS diamond drill hole database contains 356 surface diamond drill holes and 63 underground drill holes. A selection of 330 holes was considered for the resource estimate (Fig. 14.1). From these, a subset of 230 holes (169 from surface and 61 from underground) cut across the mineralized zones. The database also contains 357 surface channel samples and 192 underground channel samples. As part of the current mandate, all holes were compiled and validated before the estimate was initiated.

All 230 holes contain lithological descriptions taken from drill core logs. The 230 drill holes cover the strike-length of the project at a variable drill spacing ranging from 5 to 50 metres (mostly below 20 m). This selection of 230 drill holes contains a total of 23,203 sampled intervals taken from 23,576.18 metres of drilled core.

In addition to the basic tables of raw data, the GEMS database includes several tables containing the calculated drill hole composites and wireframe solid intersections required for the statistical evaluation and resource block modelling.

InnovExplo's data verification included a site visit to the Fenelon Camp and a review of the logging and core storage facilities. It also included a review of selected core intervals, drill hole collar locations, assays, the QA/QC program, downhole surveys, information on mined-out areas, and the descriptions of lithologies, alteration and structures. InnovExplo was able to collect and send to the laboratory eight (8) drill core quarter-splits and one (1) mineralized sample from the ore pad. Wallbridge had not yet carried out any work on the property at the time this resource estimate was being prepared.

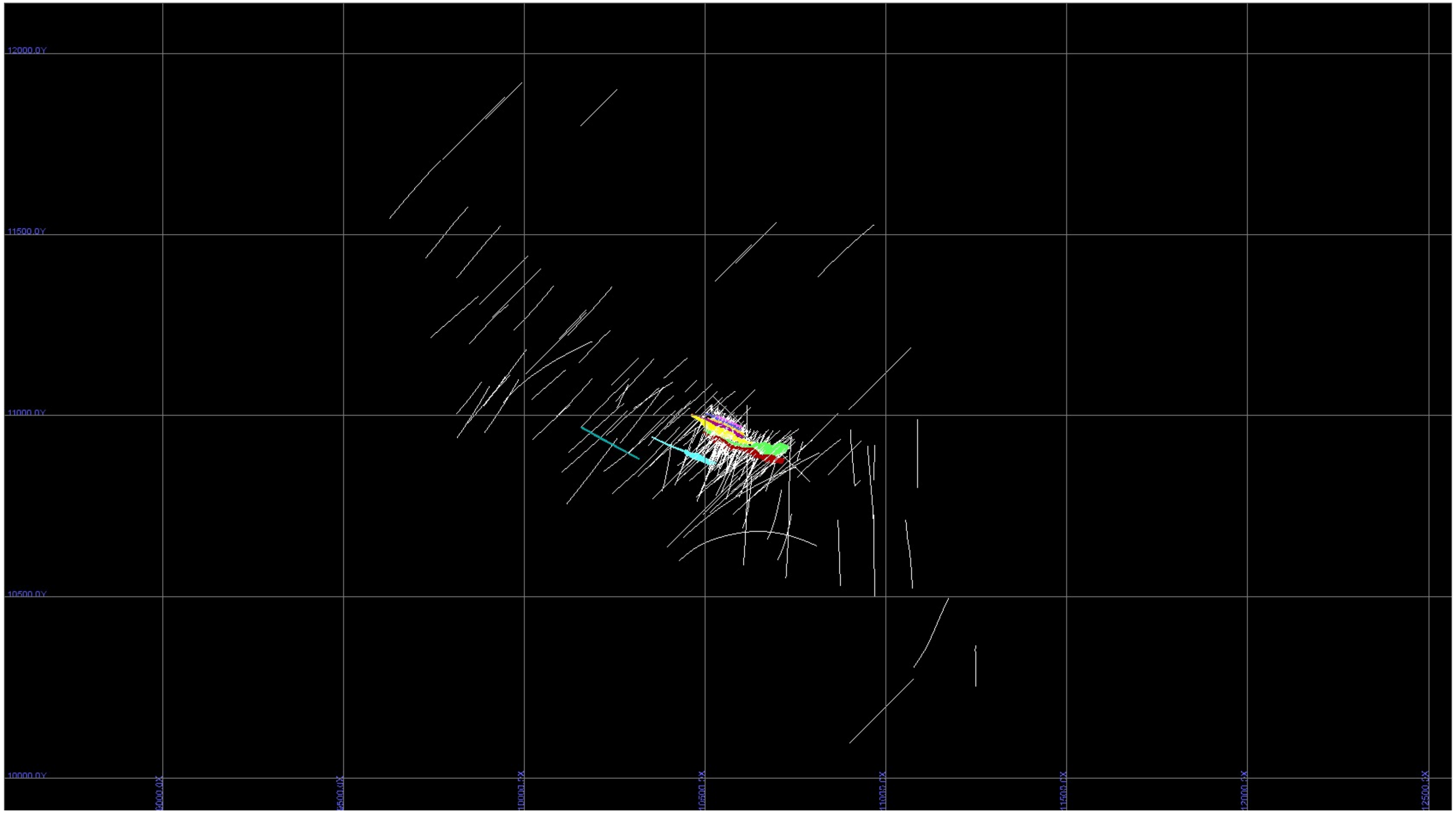


Figure 14.1 – Surface plan view of the Fenelon drill hole database used for the resource estimate (n = 330). Coloured shapes are the mineralized zones

14.2 Interpretation of Mineralized Zones

In order to conduct accurate resource modelling of the deposit, the mineralized-zone wireframe model was based on the drill hole database and the author's knowledge of the Fenelon mine and similar deposits. The model comprises nine (9) mineralized solids (coded 102 to 110) that honour the drill hole database. A total of 851 construction lines were created (154 3D rings and 697 tie lines), all of which were snapped to drill hole intercepts to produce valid solids.

Two surfaces were also created to define topography and overburden. These surfaces were generated from drill hole descriptions and survey information provided by Wallbridge.

Figure 14.2 presents a 3D view of the mineralized solids.

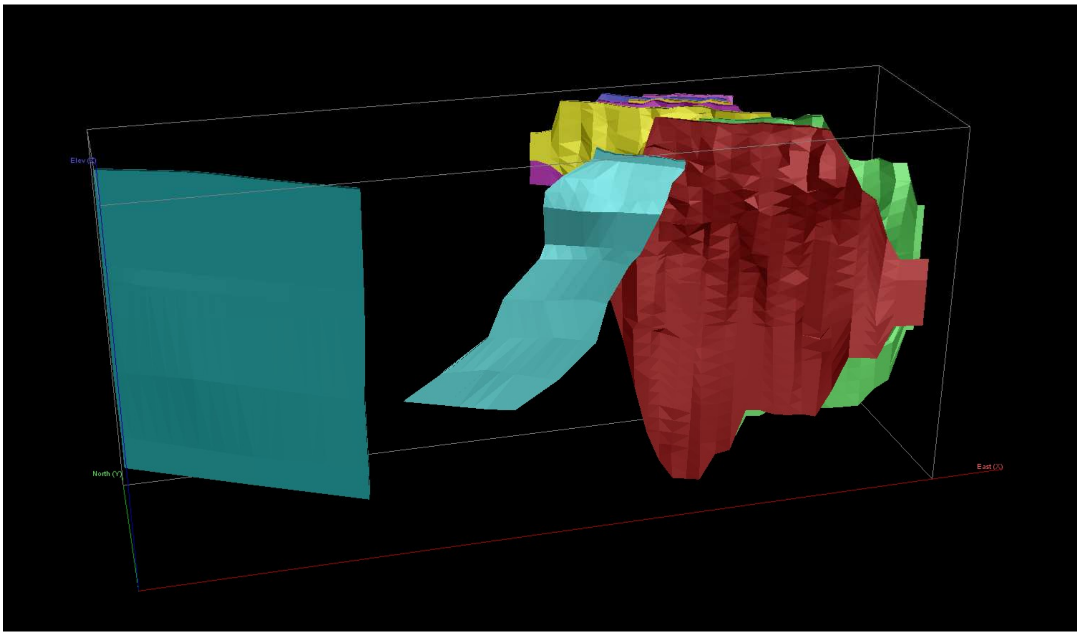


Figure 14.2 – 3D view of the mineralized model for the Fenelon deposit, looking north-northeast

14.3 Voids Model

Wallbridge provided InnovExplo with data on underground voids.

Most of the voids were already available in 3D, modelled by previous owners, but some stope contours were modelled by the Wallbridge team. Those voids had never been converted into 3D format due to the abrupt closures of the mine in the mid-2000s. They were modelled using the data from underground mapping.

Based on the available data, the voids (drifts and stopes) in GEMS project are considered accurate. Figure 14.3 shows the voids used to deplete the current resource estimate.

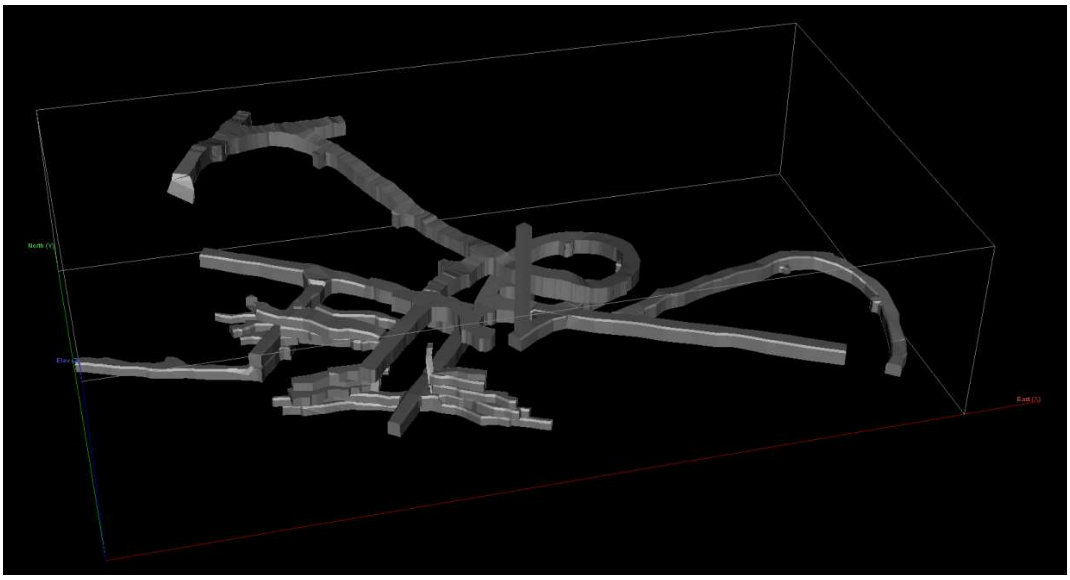


Figure 14.3 – Underground Voids used to deplete the current mineral resource estimate, looking north-northeast

Historical open pit extraction was taken into consideration in the model by merging it with the original topography and bedrock models.

Resource depletion for both extraction methods (open pit and underground) was therefore applied in the block model.

14.4 High Grade Capping

For drill hole assay intervals that intersect the interpreted mineralized zones, codes were automatically attributed based on the name of the 3D solids, and these coded intercepts were used to analyze sample lengths and generate statistics for high-grade capping and composites.

Basic univariate statistics were performed on individual raw gold assay datasets for mineralized zones 101 to 110.

The following criteria were used to decide whether capping was warranted or not, and to determine the threshold when warranted:

- If the quantity of metal contained in the last decile is above 40%, capping is warranted; if below 40%, the uncapped dataset may be used;
- No more than 10% of the overall contained metal must be contained within the first 1% of the highest grade samples;
- The probability plot of grade distribution must not show abnormal breaks or scattered points outside of the main distribution curve;
- The log normal distribution of grades must not show any erratic grade bins nor distanced values from the main population.

Table 14.1 presents a summary of the statistical analysis for each dataset. Figures 14.4 to 14.12 show graphs supporting the capping threshold decisions for all individual zones.

Table 14.1 – Summary statistics for the raw assays by dataset

Dataset	Block Code	Metal	# of Samples	Max (g/t)	Uncut Mean (g/t)	High Grade Capping (g/t)	Cut Mean (g/t)	# of Samples Cut	% of Samples Cut	% Metal Factor Loss	Coefficient of Variation
Mineralized Zone 2	102	Au (g/t)	76	93.30	4.13	30.00	2.84	3	3.95%	27.73%	2.60
Mineralized Zone 3	103	Au (g/t)	178	603.82	25.66	140.00	16.72	11	6.18%	37.91%	2.41
Mineralized Zone 4	104	Au (g/t)	164	839.55	25.21	140.00	12.23	5	3.05%	52.20%	2.66
Mineralized Zone 5	105	Au (g/t)	281	612.73	14.67	140.00	10.70	6	2.14%	15.78%	2.65
Mineralized Zone 6	106	Au (g/t)	416	897.00	14.62	140.00	8.01	10	2.40%	35.44%	3.27
Mineralized Zone S1	107	Au (g/t)	387	530.00	10.42	140.00	7.43	11	2.84%	21.45%	3.43
Mineralized Zone S2	108	Au (g/t)	294	175.87	3.35	30.00	1.71	6	2.04%	45.23%	2.92
Mineralized Zone A	109	Ag (g/t)	37	42.80	3.55	30.00	2.93	2	5.41%	17.79%	2.77
Mineralized Zone B	110	Au (g/t)	13	48.56	6.25	30.00	4.83	1	7.69%	16.26%	1.76

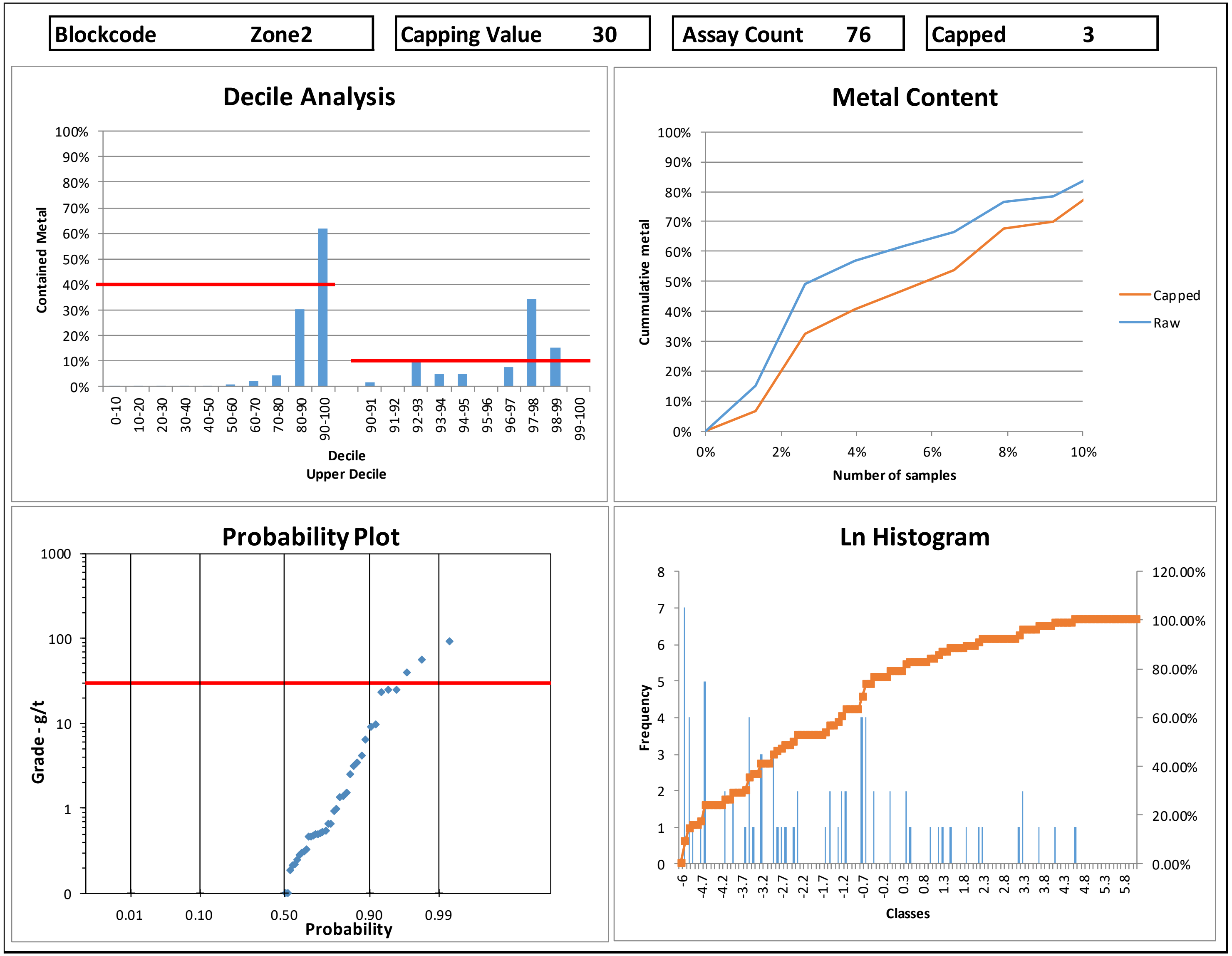


Figure 14.4 – Graphs supporting a capping grade of 30 g/t Au for mineralized zone 2

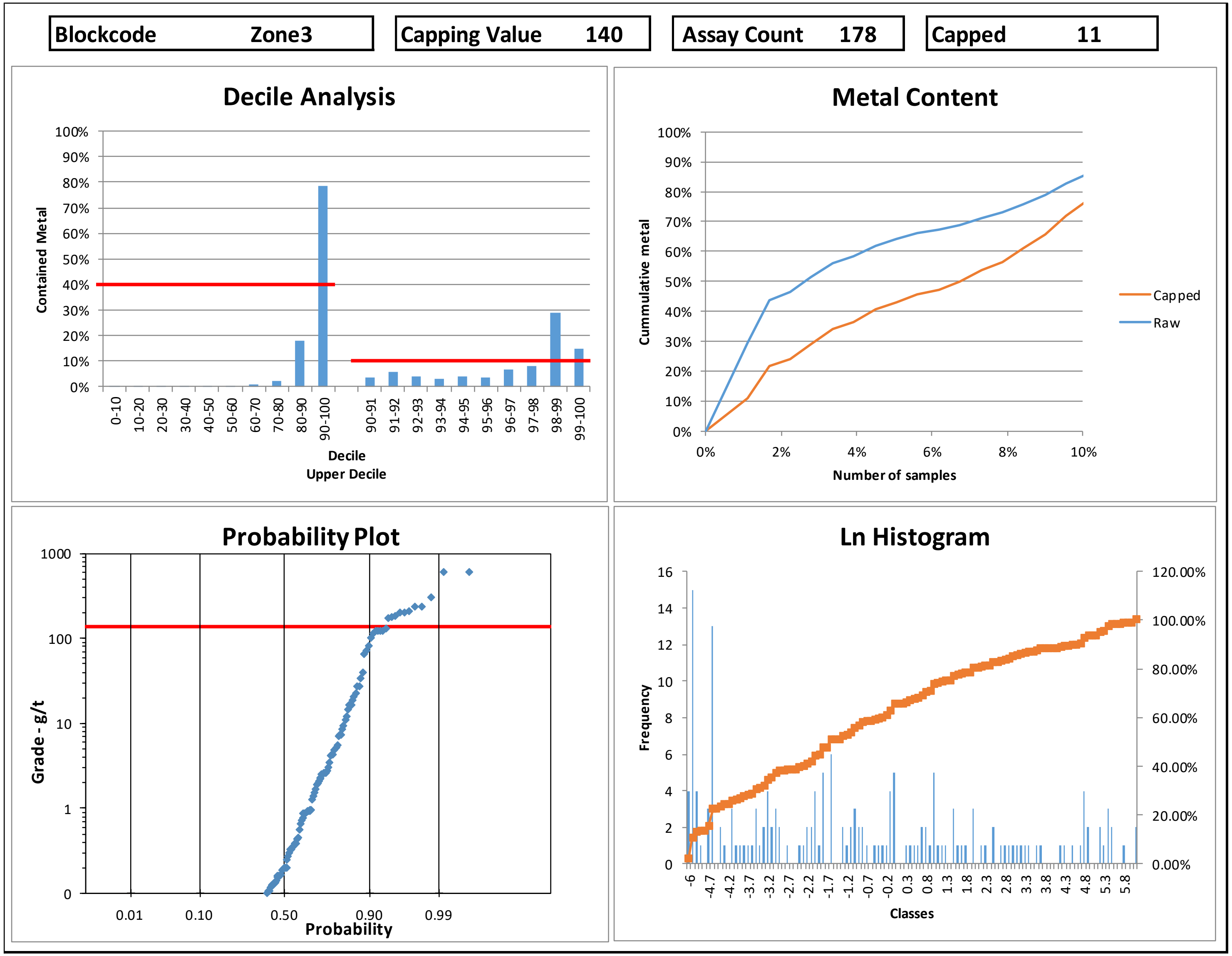


Figure 14.5 – Graphs supporting a capping grade of 140 g/t Au for mineralized zone 3

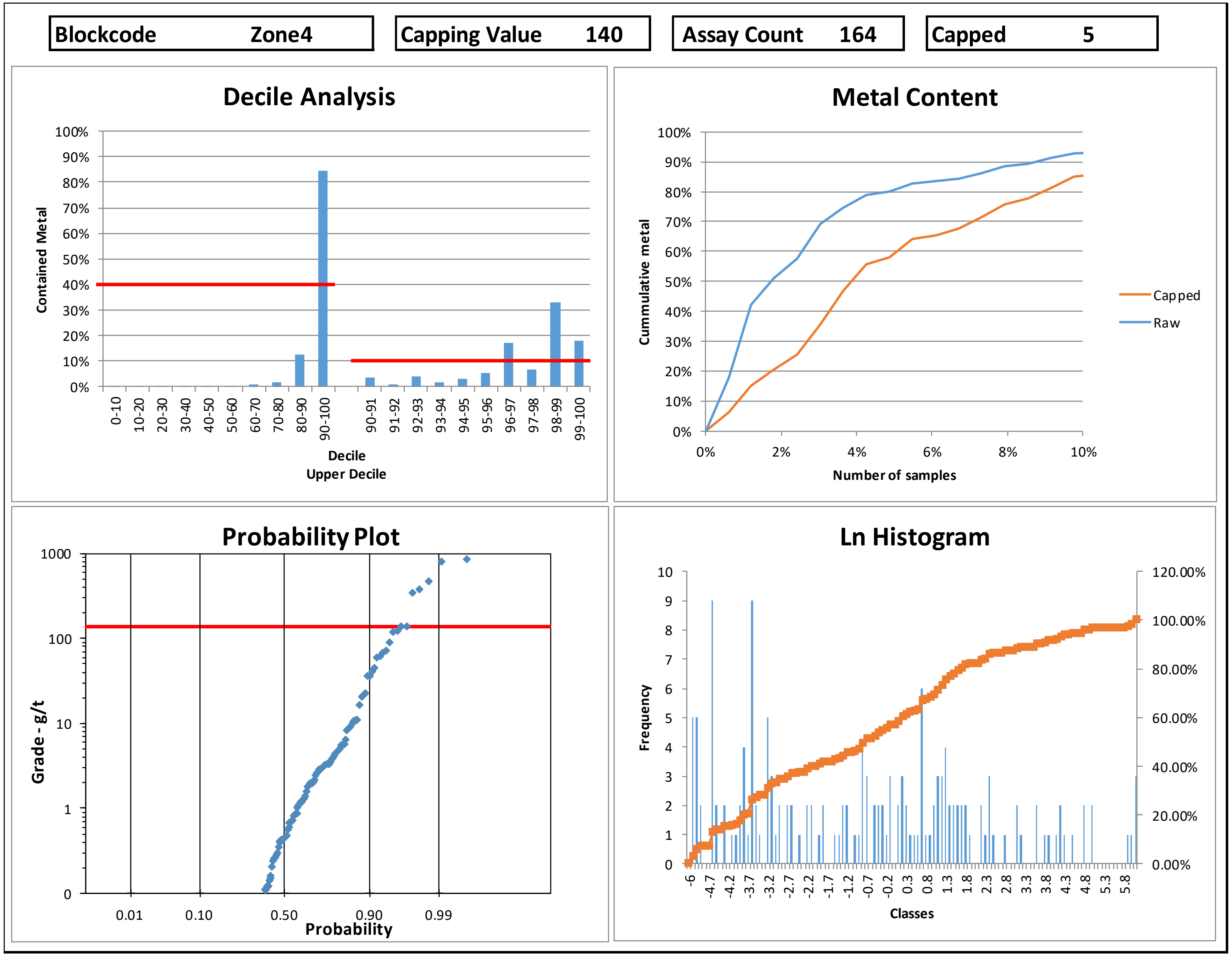


Figure 14.6 – Graphs supporting a capping grade of 140 g/t Au for mineralized zone 4

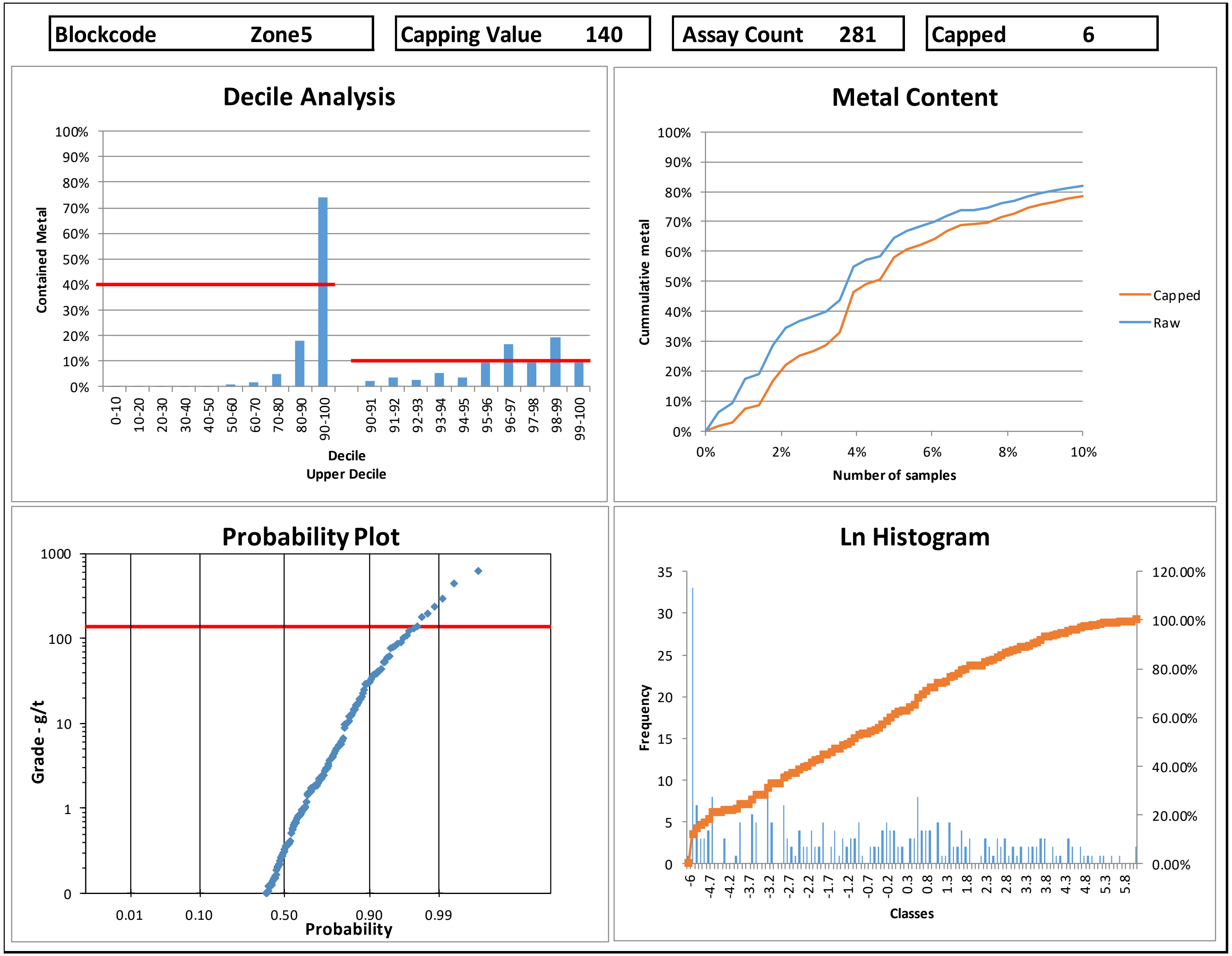


Figure 14.7 – Graphs supporting a capping grade of 140 g/t Au for mineralized zone 5

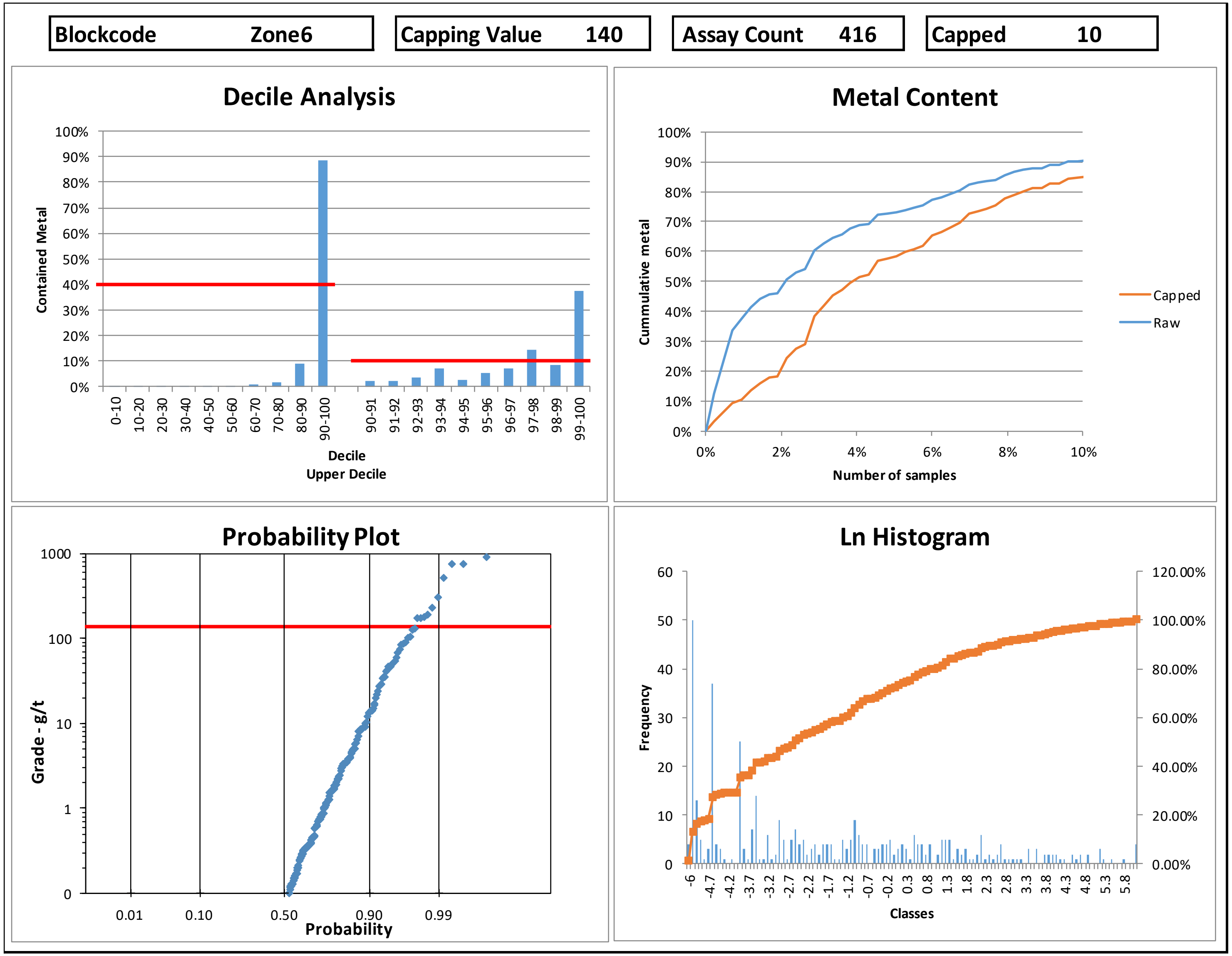


Figure 14.8 – Graphs supporting a capping grade of 140 g/t Au for mineralized zone 6

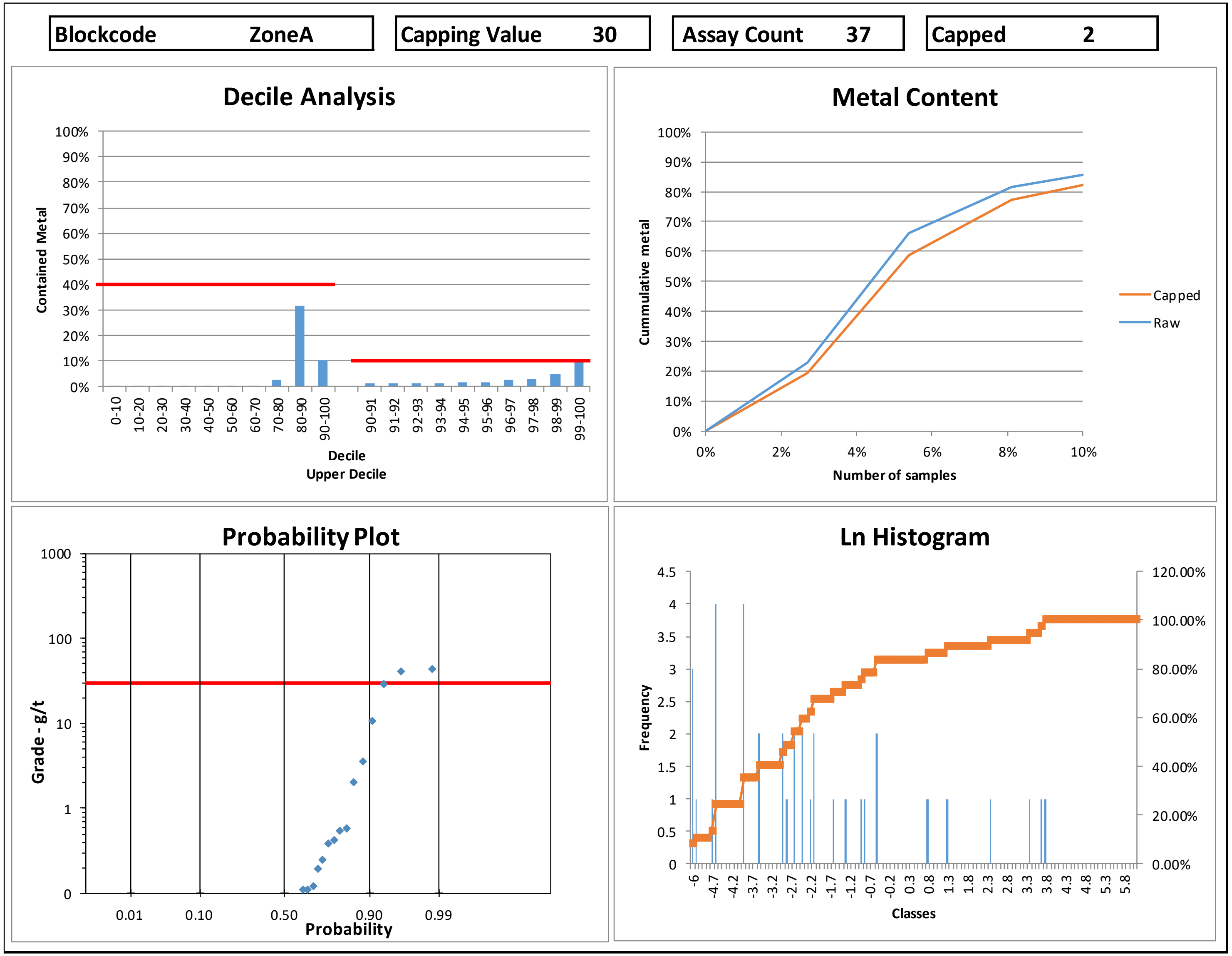


Figure 14.9 – Graphs supporting a capping grade of 30 g/t Au for mineralized zone A

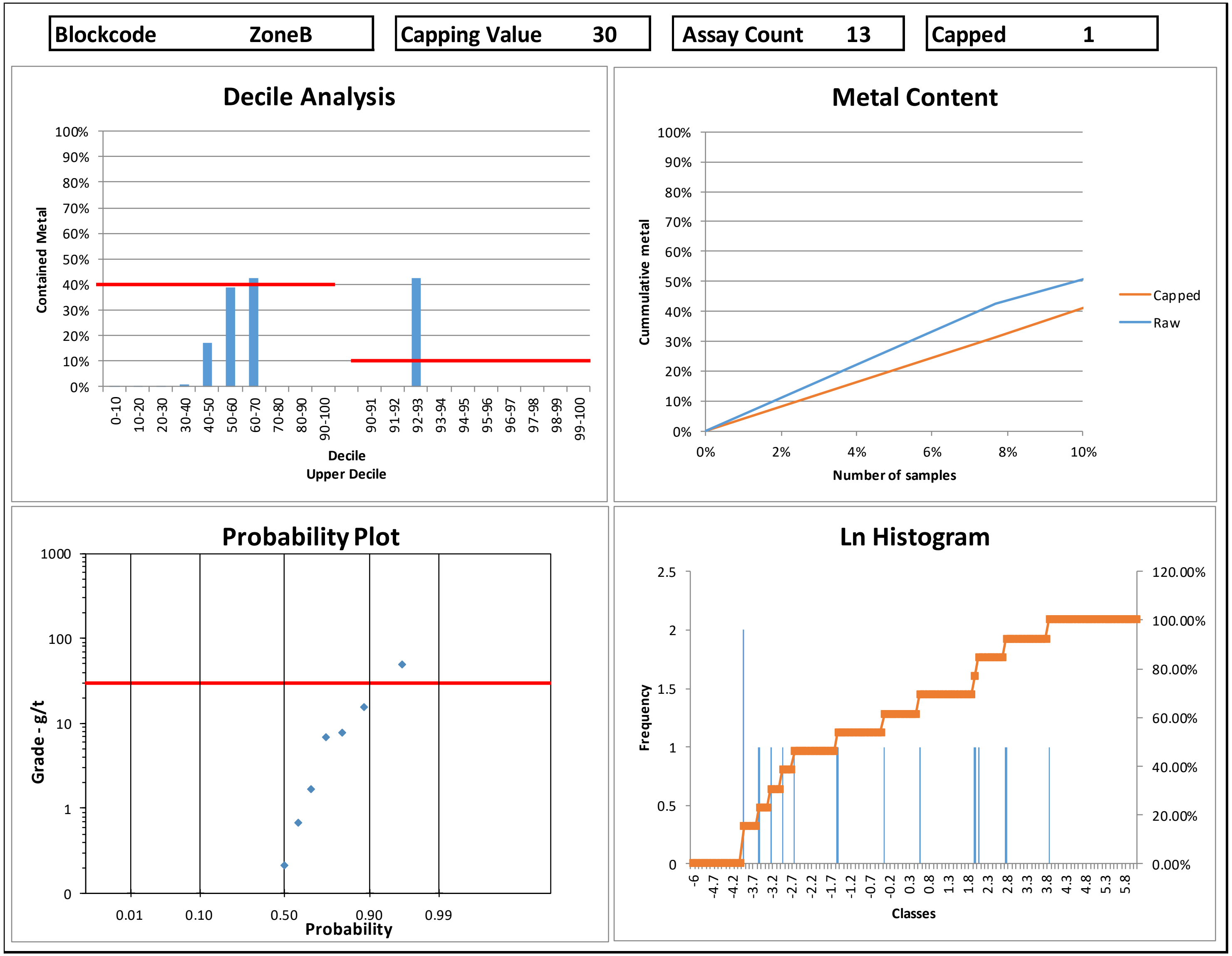


Figure 14.10 – Insufficient samples; a capping grade of 30 g/t Au was attributed to mineralized zone B

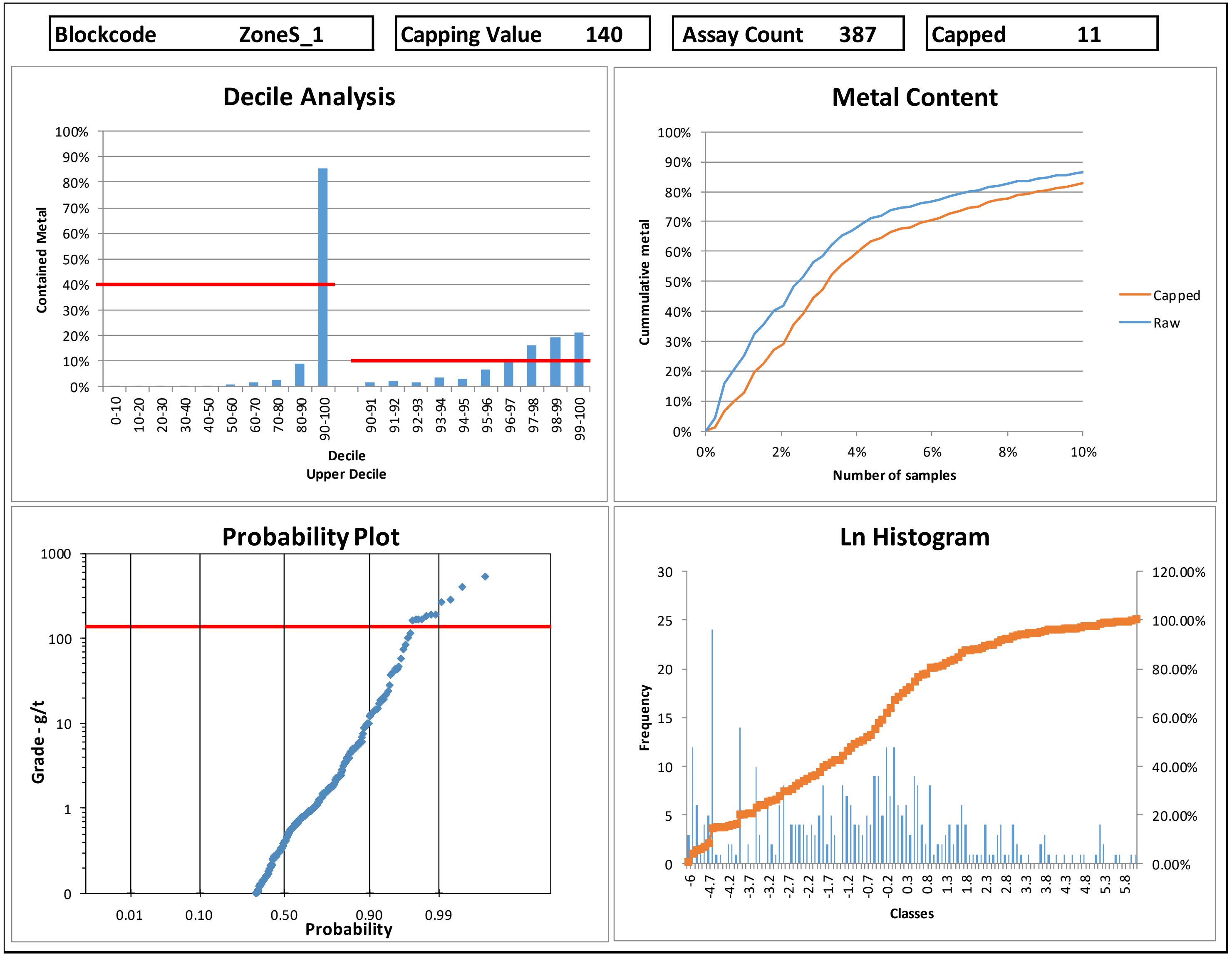


Figure 14.11 – Graphs supporting a capping grade of 140 g/t Au for mineralized zone S1

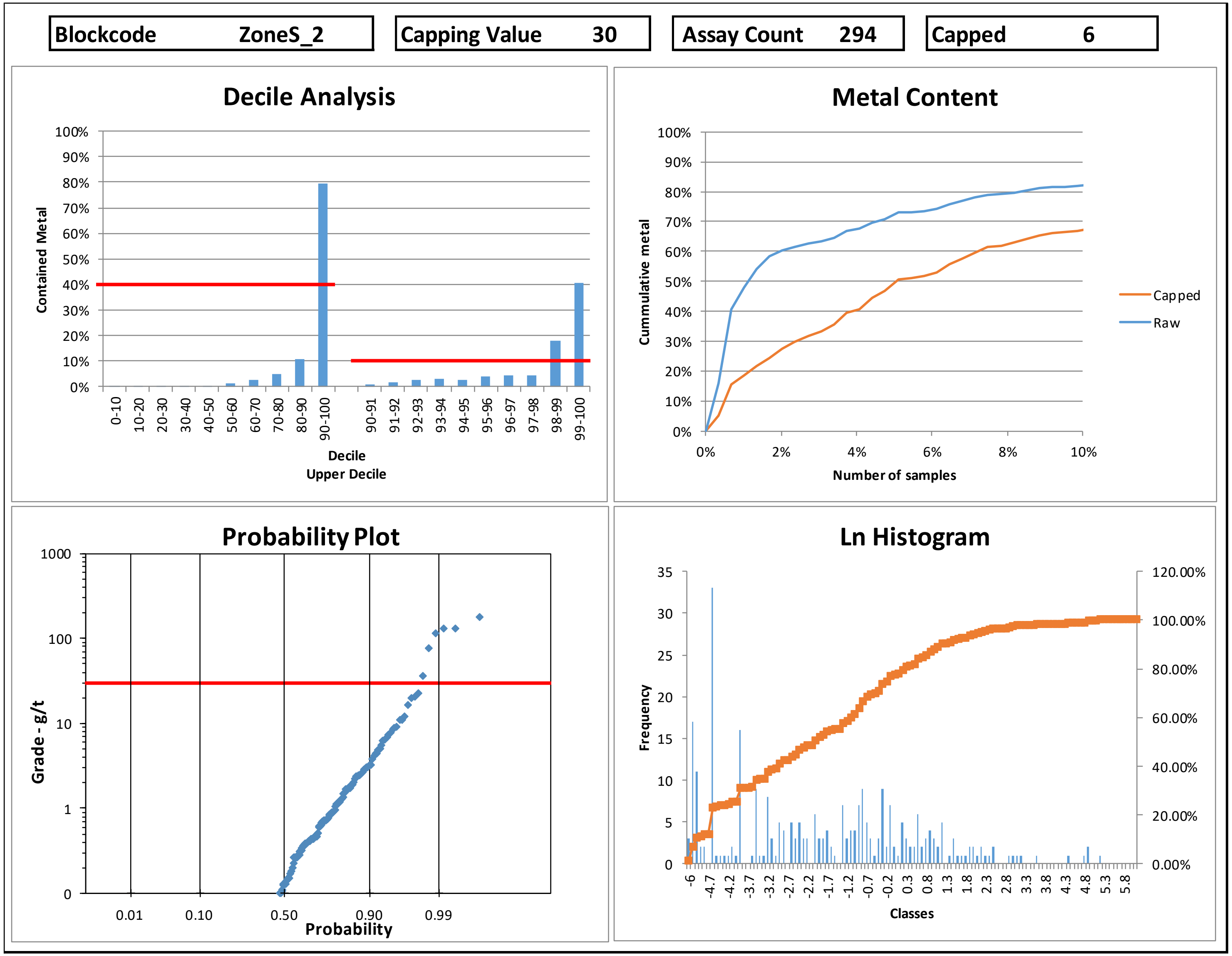


Figure 14.12 – Graphs supporting a capping grade of 30 g/t Au for mineralized zone S2

14.5 Compositing

In order to minimize any bias introduced by the variable sample lengths, the capped assays of the DDH data were composited.

A significant portion of the samples in the database are longer than 1.0 metre, mostly 1.5 metre (Fig 14.13). Using 1-metre intervals would work against the idea of compositing. And with most zones being 2 metres thick, 1.5-metre composites would be illogical as it would systematically give significant extra weight to the tails. For geological reasons, a 2-metre ("2m") composite, with an allowable spread of 1 to 3 metres, was selected as the logical option for the Fenelon deposit. This option is also supported by statistical analysis (Table 14.2). The total number of composites used in the DDH dataset is 1,294. A grade of 0.00 g/t Au was assigned to missing sample intervals. Table 14.3 shows the basic statistics for composites by zone.

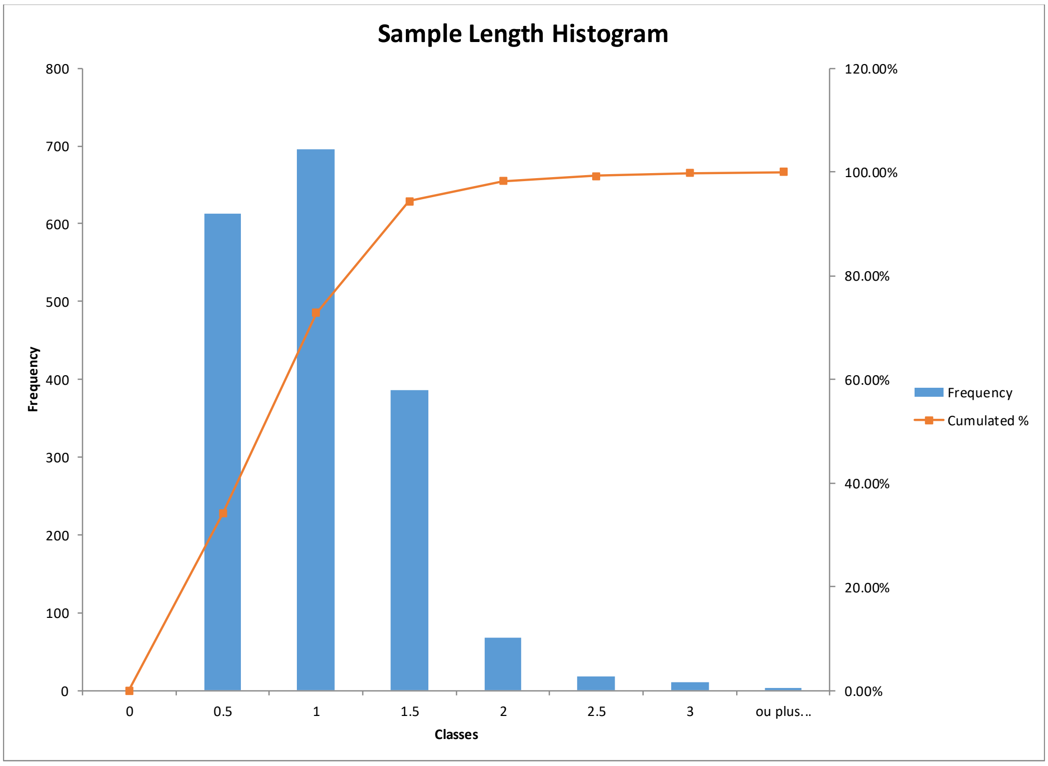


Figure 14.13 – Graphs illustrating sample length distribution within mineralized zones

Table 14.2 – Statistics supporting the choice of 2m composites with distributed tails

Population	Total Length	Accuracy	Min	Max	Ratio	Average	Mediane	WEIGTH COV	COV AU
1.0M (>0.25m)	2572.67	99.28%	0.25	1.00	4.00	0.88	1.00	0.21	3.94
1.5M (>0.25m)	2580.29	99.58%	0.25	1.50	6.00	1.25	1.50	0.28	3.71
2.0M (>0.25m)	2582.25	99.65%	0.25	2.00	8.00	1.56	2.00	0.36	3.66
2.0M Distributed (1-3)	2586.22	99.81%	1.00	3.00	3.00	2.00	1.93	0.18	3.39

Intercepts	2591.25	100.00%

Table 14.3 – Summary statistics for the composites

Dataset	Block Code	Metal	# of Composites	Max (g/t)	Mean (g/t)	Standard Deviation	Coefficient of Variation
Mineralized Zone 2	102	Au (g/t)	46	14.09	1.24	3.04	2.45
Mineralized Zone 3	103	Au (g/t)	132	102.64	6.88	17.51	2.55
Mineralized Zone 4	104	Au (g/t)	128	139.40	6.10	19.38	3.18
Mineralized Zone 5	105	Au (g/t)	180	102.07	4.85	13.87	2.86
Mineralized Zone 6	106	Au (g/t)	284	99.28	4.14	13.46	3.25
Mineralized Zone S1	107	Au (g/t)	280	135.76	3.77	13.19	3.50
Mineralized Zone S2	108	Au (g/t)	212	20.66	0.79	2.19	2.77
Mineralized Zone A	109	Ag (g/t)	24	14.98	1.70	3.61	2.12
Mineralized Zone B	110	Au (g/t)	8	13.26	3.99	4.36	1.09

14.6 Density

Densities are used to calculate tonnages from the volume estimates in the resource-grade block model.

The author's usual approach is to compare all available data to establish what can be used. In Fenelon's case, only the following limited information is available:

- PAH used a density of 2.70 in 2001 (GM60703), which was the density used historically by Taurus at the time. There was no data to support this value.
- A 20-kilogram core composite sample yielded a density value of 2.823 g/cm^3 at the *Centre de Recherche Minérale of Ste-Foy*, as reported by SRK in 2003 (GM60704).
- A value of 2.80 g/cm^3 seems to have been used during mining in 2004. No data was found to support this value.
- Following the site visit in May 2016, Wallbridge sent seven (7) samples to the laboratory that ran 2.78 g/cm^3 to 2.97 g/cm^3 (average 2.88 g/cm^3; median 2.90 g/cm^3).
- Following the site visit in May 2016, InnovExplo sent nine (9) samples to the laboratory that ran 2.54 g/cm^3 to 2.80 g/cm^3 (average 2.68 g/cm^3; median 2.69 g/cm^3).

Based on this limited information, InnovExplo recommends using a fixed density value of 2.80 g/cm^3, which represents the average of the three pertinent values provided above. PAH's value of 2.70 g/cm^3 was discarded due to the apparent lack of supporting information.

14.7 Block Model

A block model was established for the purpose of the current resource estimate. The block model covers an area sufficient to host an open pit, if necessary. The model has been pushed down to a depth of approximately 300 metres below surface. The block model was rotated. The block dimensions reflect the sizes of the mineralized zones and plausible mining methods. Table 14.4 provides the properties of the block model.

Table 14.4 – Block model properties

Properties	X (Columns)	Y (Rows)	Z (Levels)
Origin coordinates (UTM NAD83)	9997.748	10873.671	5280
Block size	5	5	5
Number of blocks	165	100	65
Block model extent (m)	825	500	325
Rotation	-26		

All blocks with more than 0.001% of their volume falling within a selected solid were assigned the corresponding solid block code in their respective folder. A percent block model was generated, reflecting the proportion of each block inside every solid (i.e., individual mineralized zones, individual lithological domains, the overburden and the country rock).

Table 14.5 provides details about the naming convention for the corresponding GEMS solids, as well as the rock codes and block codes assigned to each individual solid. The multi-folder percent block model thus generated was used for the mineral resource estimation.

Table 14.5 – Block model naming convention and codes

Workspace	Description	Rockcode	GEMS Triangulation Name			Precedence
			NAME1	NAME2	NAME3	
Zones_A	Mineralized Zone 2	102	Zone2	Final_Clip	F160626	10
	Mineralized Zone 2	104	Zone4	Final_Clip	F160626	12
	Mineralized Zone 2	106	Zone6	Final_Clip	F160705	14
	Mineralized Zone 2	123, 124	ZoneS_2	Final_Clip	F160626	18
	Mineralized Zone 2	109	ZoneA	Final_Clip	F160626	15
	Mineralized Zone 2	110	ZoneB	Final_Clip	F160626	16
Zones_B	Mineralized Zone 2	103	Zone3	Final_Clip	F160626	11
	Mineralized Zone 2	105	Zone5	Final_Clip	F160705	13
	Mineralized Zone 2	121, 122	ZoneS_1	Final_Clip	F160626	17
Voids	Underground infrastructures	25	Solid	Voids	F160626	1
Waste	Overburden and air	50	Surface	Topo_2016	F160626	0
	Country Rock	999	Surface	Topo_2016	F160626	0

14.8 Variography and Search Ellipsoids

Three-dimensional directional variography was completed on DDH composites of the capped gold assay data for all individual mineralized zones. The study was carried out in the software Supervisor. The 3D directional-specific investigations yielded the best-fit model along an orientation that corresponds to the strike and dip of the mineralized zones.

For most zones, the data does not allow for a nugget effect to be established from downhole variograms due to the fact that not enough samples are found within individual intercepts (2 to 3 m thick). When all zones are combined, the downhole variogram suggests a nugget effect of 0.10 (Fig. 14.14). This value was used for all zones.

Figure 14.15 shows an example of the variography study for Zone 106.

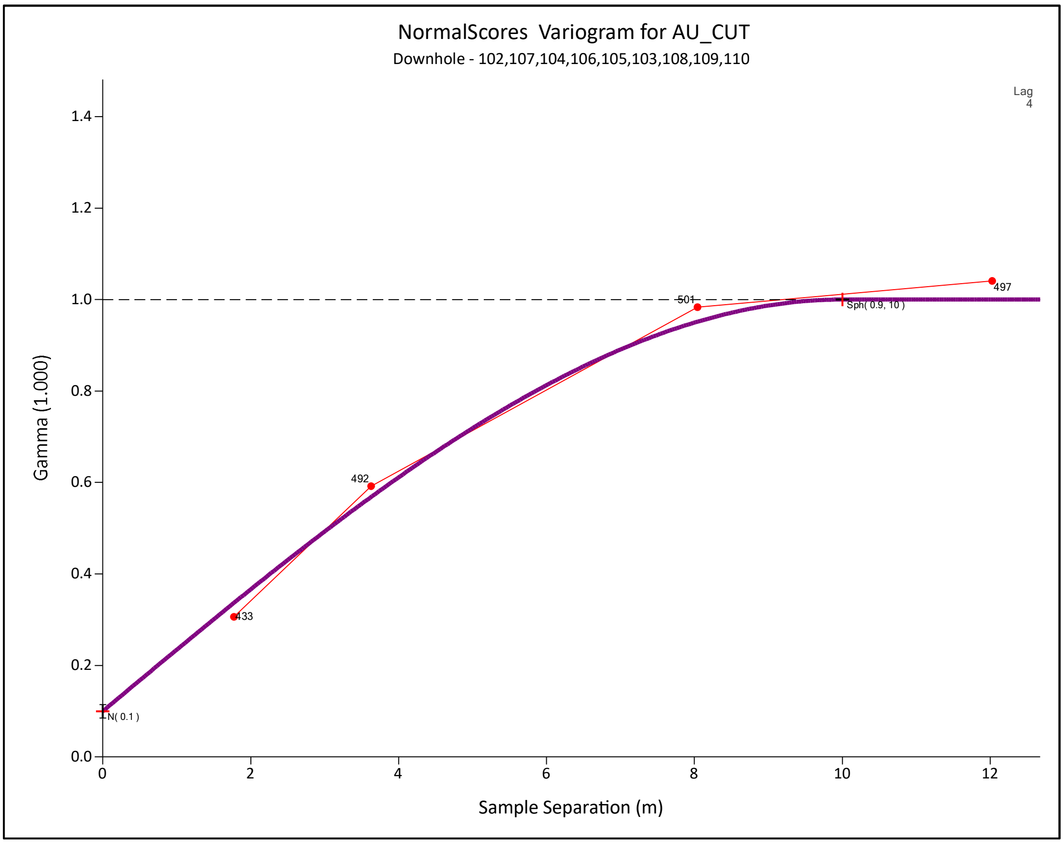


Figure 14.14 – Graph showing the nugget effect value of 0.10 derived from the variography study.

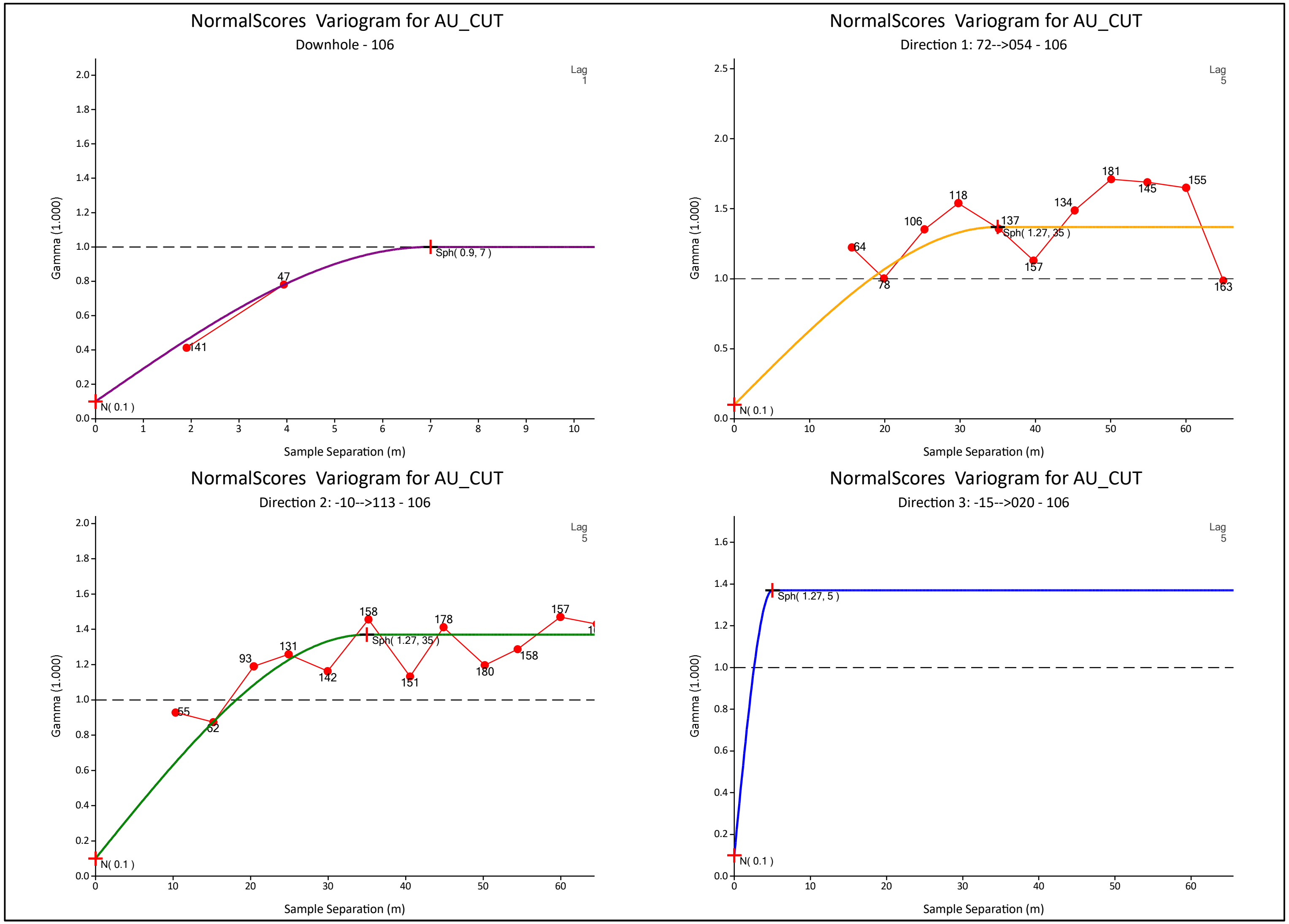


Figure 14.15 – Example of variography study for Zone 106

Two ellipsoids were built from the results of the variography study. These correspond to: a) the variography results; and b) twice the variography results. Figure 14.16 shows the variography ellipsoid for Zone 106 on a longitudinal view.

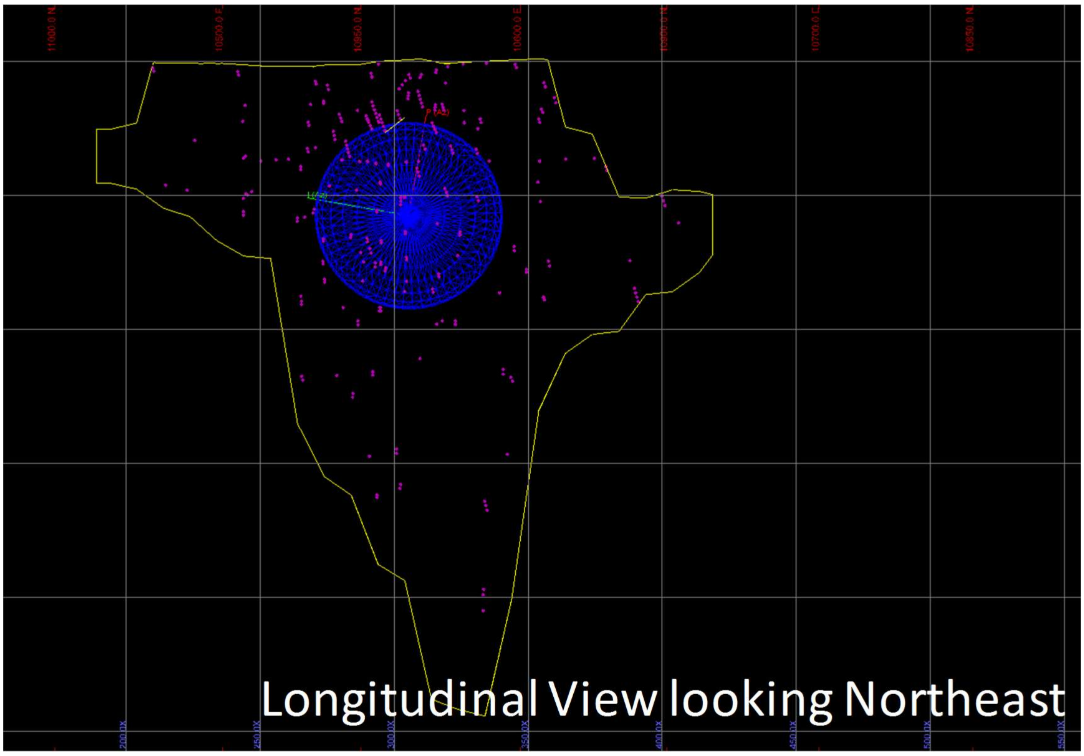


Figure 14.16 – Longitudinal view of Zone 106, looking northeast, showing the ellipsoid obtained from the variography study.

Tables 14.6 summarizes the parameters of the final ellipsoids used for the interpolation.

Table 14.6 – Search ellipsoid parameters

Zone	Blockcode	Ellipsoid	ORIENTATION			RANGES			General Parameters			Restricted Search Ellipsoid			
			Z (Gems)	X (Gems)	Z (Gems)	X (m)	Y (m)	Z (m)	Min Composites	Max Composites	Minimum DDH	X (m)	Y (m)	Z (m)	Threshold (g/t)
Mineralized Zone 2	102	P1	1	85	90	15	15	5	3	9	2	-	-	-	-
		P2	1	85	90	30	30	10	3	9	1	-	-	-	-
Mineralized Zone 3	103	P1	1	85	90	20	20	5	3	9	2	-	-	-	-
		P2	1	85	90	40	40	10	3	9	1	20	20	5	30
Mineralized Zone 4	104	P1	1	80	80	35	35	5	3	9	2	-	-	-	-
		P2	1	80	80	70	70	10	3	9	1	35	35	5	30
Mineralized Zone 5	105	P1	1	80	80	30	30	5	3	9	2	-	-	-	-
		P2	1	80	80	60	60	10	3	9	1	30	30	5	30
Mineralized Zone 6	106	P1	3	85	80	35	35	5	3	9	2	-	-	-	-
		P2	3	85	80	70	70	10	3	9	1	35	35	5	30
Mineralized Zone S1	107E	P1	21	80	80	40	40	5	3	9	2	-	-	-	-
		P2	21	80	80	80	80	10	3	9	1	40	40	5	30
Mineralized Zone S1	107W	P1	1	85	80	40	40	5	3	9	2	-	-	-	-
		P2	1	85	80	80	80	10	3	9	1	40	40	5	30
Mineralized Zone S2	108E	P1	11	85	80	55	55	5	3	9	2	-	-	-	-
		P2	11	85	80	110	110	10	3	9	1	-	-	-	-
Mineralized Zone S2	108W	P1	1	85	80	55	55	5	3	9	2	-	-	-	-
		P2	1	85	80	110	110	10	3	9	1	-	-	-	-
Mineralized Zone A	109	P1	1	90	80	15	15	5	3	9	2	-	-	-	-
		P2	1	90	80	30	30	10	3	9	1	-	-	-	-
Mineralized Zone B	110	P1	-4	90	80	15	15	5	3	9	2	-	-	-	-
		P2	-4	90	80	30	30	10	3	9	1	-	-	-	-

14.9 Grade Interpolation

The variography study provided the parameters to interpolate the grade model using composites from the capped grade data in order to produce the best possible grade estimate for the defined resources. The interpolation was run on a point area workspace extracted from the DDH dataset.

The composite points were assigned block codes corresponding to the mineralized zone in which they occur. The interpolation profiles specify a single composite block code for each mineralized-zone solid, thus establishing hard boundaries between the mineralized zones and preventing block grades from being estimated using sample points with different block codes than the block being estimated.

The interpolation profiles were customized to estimate grades separately for each of the mineralized zones. Four interpolation methods were investigated (ID2, ID3, OK, NN). The inverse distance to the third power (ID3) method was selected for the final resource estimation as it better honours the Fenelon deposit grade distribution.

Two passes were defined. The ellipsoid radiuses from Pass 1 were established using the variography results. Ellipsoid radiuses from Pass 2 were twice the variography results. Pass 2 interpolated only blocks that were not interpolated during Pass 1. A restricted search ellipsoid on high-grade composites was also applied to Pass 2 in order to limit grades higher than 30 g/t within the variography range.

Parameters used to interpolate gold during Pass 1:

- Variography ranges results;
- Minimum 2 holes;
- Minimum 3 composites;
- Maximum 9 composites.

Parameters used to interpolate gold during Pass 2:

- Twice variography ranges results;
- Minimum 3 composites;
- Maximum 9 composites;
- Restricted search ellipsoid on >30 g/t Au composites using variography ranges

14.10 Resource Categories

14.10.1 Mineral resource classification definition

The resource classification definitions used for this report are those published by the Canadian Institute of Mining, Metallurgy and Petroleum in their document "*CIM Definition Standards for Mineral Resources and Reserves*".

Measured Mineral Resource: that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.

Indicated Mineral Resource: that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

Inferred Mineral Resource: that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

14.10.2 Mineral resource classification

All interpolated blocks were assigned to the Inferred category during the creation of the grade block model to make sure that sufficient continuity was observed in order to avoid isolated blocks being interpolated by only one hole.

The reclassification to an Indicated category was done for blocks meeting all the conditions below:

- Blocks showing geological and grade continuity;
- Blocks from well defined mineralized zones only;
- Blocks from Pass 1 only;
- Blocks interpolated by a minimum of two holes; and
- Blocks for which the distance to the closest composite is less than 20 metres.

The reclassification to a measured category was done for blocks meeting all the conditions below:

- Blocks showing geological and grade continuity;
- Blocks from well defined mineralized zones only;
- Blocks from Pass 1 only;
- Blocks interpolated by a minimum of two holes;
- Blocks classified as Indicated as per above stated conditions;
- Blocks for which the distance to the closest composite is less than 20 metres;
- Blocks for which the distance to the closest drift is less than 10 metres.

A series of outline rings (clipping boundaries) were created in long views using the criteria described above, but also keeping in mind that a significant cluster of blocks is necessary to obtain a resource. Within the Indicated resource outlines, some Inferred blocks were upgraded to the Indicated category, whereas outside these outlines, some Indicated blocks were downgraded to the Inferred category. The author is of the opinion that this was a necessary step to homogenize (smooth out) the resource volumes in each category, and to avoid isolated blocks from being included in the Indicated and Measured categories.

14.11 Cut-off Grade

The selected cut-off of 5 g/t was used to determine the mineral potential of the Fenelon deposit. The determination of the cut-off grade (CoG) was based on the parameters presented in Table 14.7.

Table 14.7 – Parameters used to estimate the cut-off grade (CoG) for the 2016 Fenelon Deposit Mineral Resource Estimate

	Exchange Rate (USD/CAD)		1.19
	Gold price (USD)	US$/oz	$ 1,225.00
Gp	Gold price (CAD)	CAD$/oz	$ 1,457.75
Pc	Processing cost	C$/t	$ 35.00
	Transport	C$/t	$ 33.00
r	Metallurgical Recovery	%	95.0%
d	Dilution for insitu Resource	%	0.0%
Gmc	Global mining cost	C$/t	$ 152.00
	Total cost by metric tonne	C$/t	$ 220.00
COG	Resource Cut-off grade	g/t Au	4.94

The gold price and exchange rate are based on the 3-year trailing average. Figure 14.17 illustrates how the metal prices and exchange rate were determined.

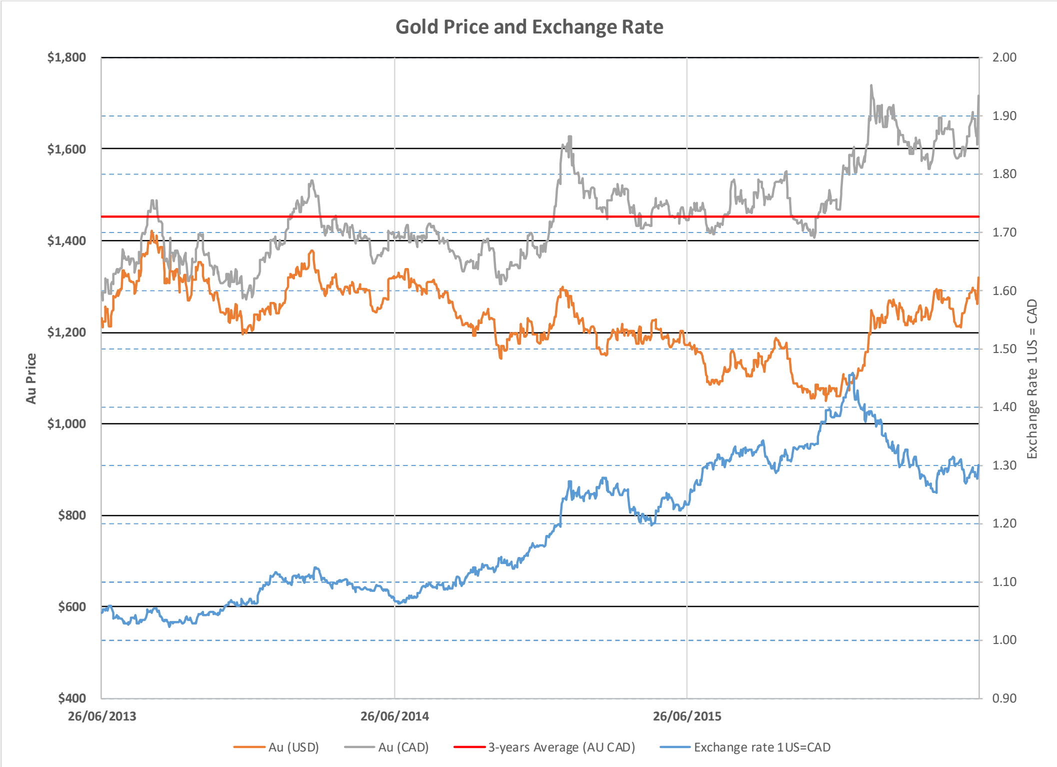


Figure 14.17 – Graph showing variations in gold price (in $US), the exchange rate and the resulting gold price in Canadian dollars.

The red line represents the value used to determine the cut-off grade for the resource estimate presented in this report (3-year average).

14.12 Mineral Resource Estimate

Given the density of the processed data, the search ellipse criteria, the drill hole density and the specific interpolation parameters, InnovExplo is of the opinion that the current mineral resource estimate can be classified as Measured, Indicated and Inferred resources. The estimate is compliant with CIM standards and guidelines for reporting mineral resources and reserves.

Table 14.8 displays the results of the In Situ Fenelon Deposit Mineral Resource Estimate at the official 5.00 g/t Au cut-off grade. Table 14.9 displays the official in-situ resource and sensitivity at other cut-off scenarios. The reader should be cautioned that the figures listed in Table 14.10 should not be misinterpreted as a mineral resource statement. The reported quantities and grade estimates at different cut-off grades are only presented to demonstrate the sensitivity of the resource model to the selection of a reporting cut-off grade. Note that broken measured resources are not included in this table since they were included in the official resource statement as a whole.

Figure 14.18 shows the grade distribution of the Fenelon deposit above the selected 5.00 g/t Au cut-off grade, and Figure 14.19 shows the category distribution above the selected 5.00 g/t Au cut-off grade.

Table 14.8 – Fenelon Deposit Mineral Resource Estimate at a 5.00 g/t Au cut-off grade

> 5.00 g/t Au		Tonnes (t)	Au (g/t)	Contained Au (oz)
Measured (M) and Indicated (I)	Measured (In-situ)	27,000	13.94	12,100
	Measured (broken)	3,100	6.14	600
	Indicated	61,000	12.89	25,300
	Total M+I	91,100	12.97	38,000
Inferred	In-situ	6,500	9.15	1,900

- The Independent and Qualified Persons for the Mineral Resource Estimate, as defined by NI 43-101, are Pierre-Luc Richard, P.Geo., M.Sc., and Catherine Jalbert, P.Geo., B.Sc., both of InnovExplo Inc. The effective date of the estimate is July 5, 2016.
- These Mineral Resources are not Mineral Reserves and thus do not have demonstrated economic viability.
- The model includes nine gold-bearing zones, eight of which include resources at the official cut-off grade.
- Results are presented in situ and undiluted.
- Sensitivity was assessed using cut-off grades from 2.00 to 10.00 g/t Au, at 1.00 g/t Au increments. The official resource is reported at a cut-off of 5.00 g/t Au. Cut-off grades must be re-evaluated in light of prevailing market conditions (gold price, exchange rate and mining cost).
- A fixed density of 2.80 g/cm^3 was used for all zones supported by limited information.
- A minimum true thickness of 2.0 metres was applied, using the grade of the adjacent material when assayed or a value of zero when not assayed.
- High grade capping (Au) was done on raw assay data and ranges from 30 g/t to 140 g/t based on the statistical analyses of individual mineralized zones. Restricted search ellipsoids were used during interpolation using 1X variography ranges and a threshold of 30 g/t Au.
- Compositing was done on drill hole intercepts falling within the mineralized zones (composite lengths vary from 1 metre to 3 metres in order to distribute the tails adequately).
- Resources were evaluated from drill holes using a 2-pass ID3 interpolation method in a block model (block size = 5 m x 5 m x 5 m).
- The inferred category is only defined within the areas where blocks were interpolated during pass 1 or pass 2 where continuity is sufficient to avoid isolated blocks being interpolated by only one drill hole. The indicated category is only defined by blocks interpolated by a minimum of two drill holes in areas where the maximum distance to the closest drill hole composite is less than 20 metres for blocks interpolated in pass 1. The measured category is only defined by blocks interpolated by a minimum of two drill holes in areas where the maximum distance to the closest drill hole composite is less than 20 metres for blocks interpolated in pass 1 and in close proximity with sampled drifts (<10 m).
- Ounce (troy) = tonnes x grade / 31.10348. Calculations used metric units (metres, tonnes and g/t).
- The number of metric tons was rounded to the nearest hundred. Any discrepancies in the totals are due to rounding effects. Rounding followed the recommendations in NI 43-101.
- InnovExplo is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, marketing or other relevant issue that could materially affect the Mineral Resource Estimate.

Table 14.9 – Fenelon Deposit Mineral Resource Estimate at a 5.00 g/t Au cut-off grade and sensitivity at other cut-off scenarios.*

	Measured			Indicated			Inferred		
Cut-off	Tonnage	Grade	Ounces	Tonnage	Grade	Ounces	Tonnage	Grade	Ounces
2.00	39,400	10.57	13,400	144,900	7.23	33,700	27,500	4.15	3,700
3.00	33,600	11.97	12,900	100,900	9.33	30,200	11,100	6.86	2,500
4.00	29,800	13.04	12,500	77,100	11.13	27,600	7,700	8.39	2,100
5.00	**27,000**	**13.94**	**12,100**	**61,000**	**12.89**	**25,300**	**6,500**	**9.15**	**1,900**
6.00	25,000	14.60	11,800	50,400	14.46	23,400	5,100	10.11	1,700
7.00	22,100	15.67	11,100	42,300	15.98	21,700	4,700	10.44	1,600
8.00	20,400	16.33	10,700	34,200	18.00	19,800	4,100	10.87	1,400
9.00	17,100	17.87	9,800	30,400	19.19	18,800	3,100	11.63	1,200
10.00	14,200	19.59	8,900	27,400	20.24	17,900	2,200	12.50	900

*Note that broken measured resources are not included in this table since they were included in the official resource statement as a whole.

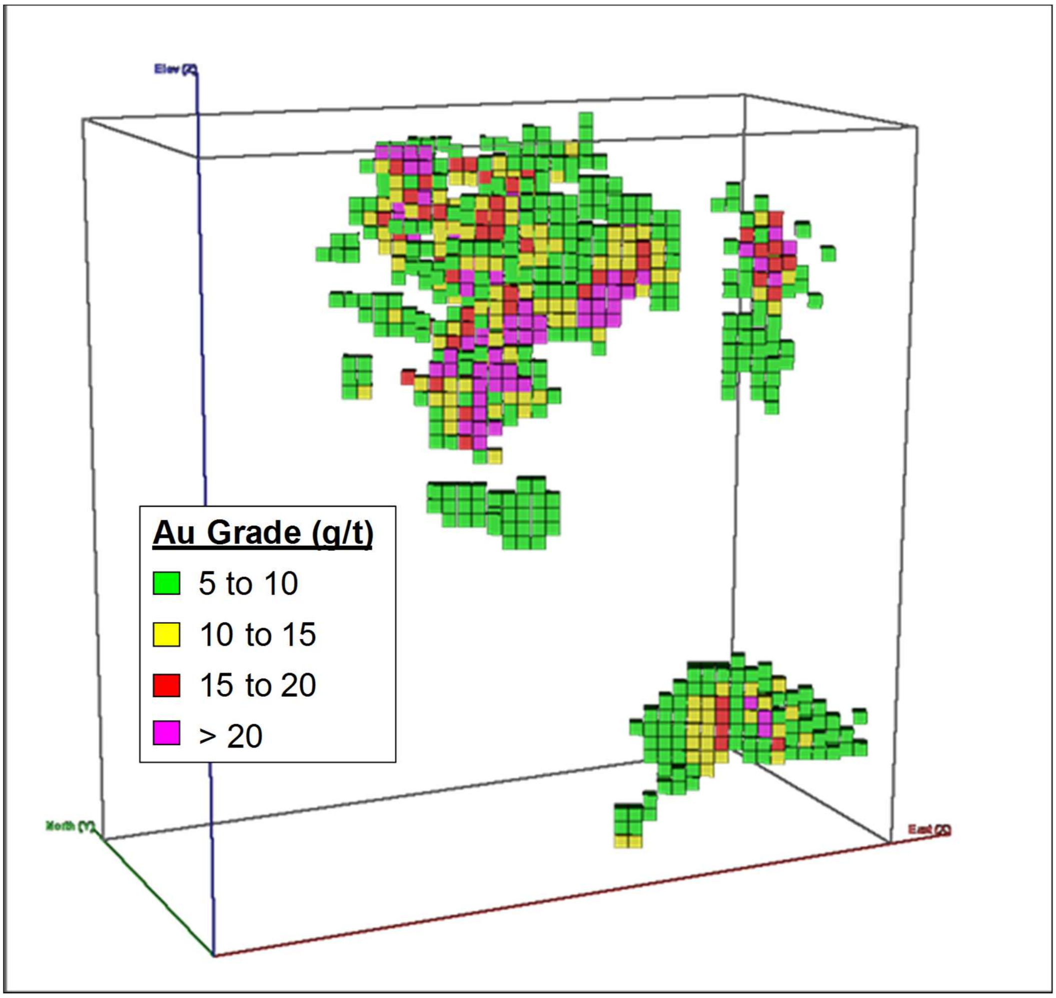


Figure 14.18 – Grade distribution above the selected 5.00 g/t Au cut-off grade

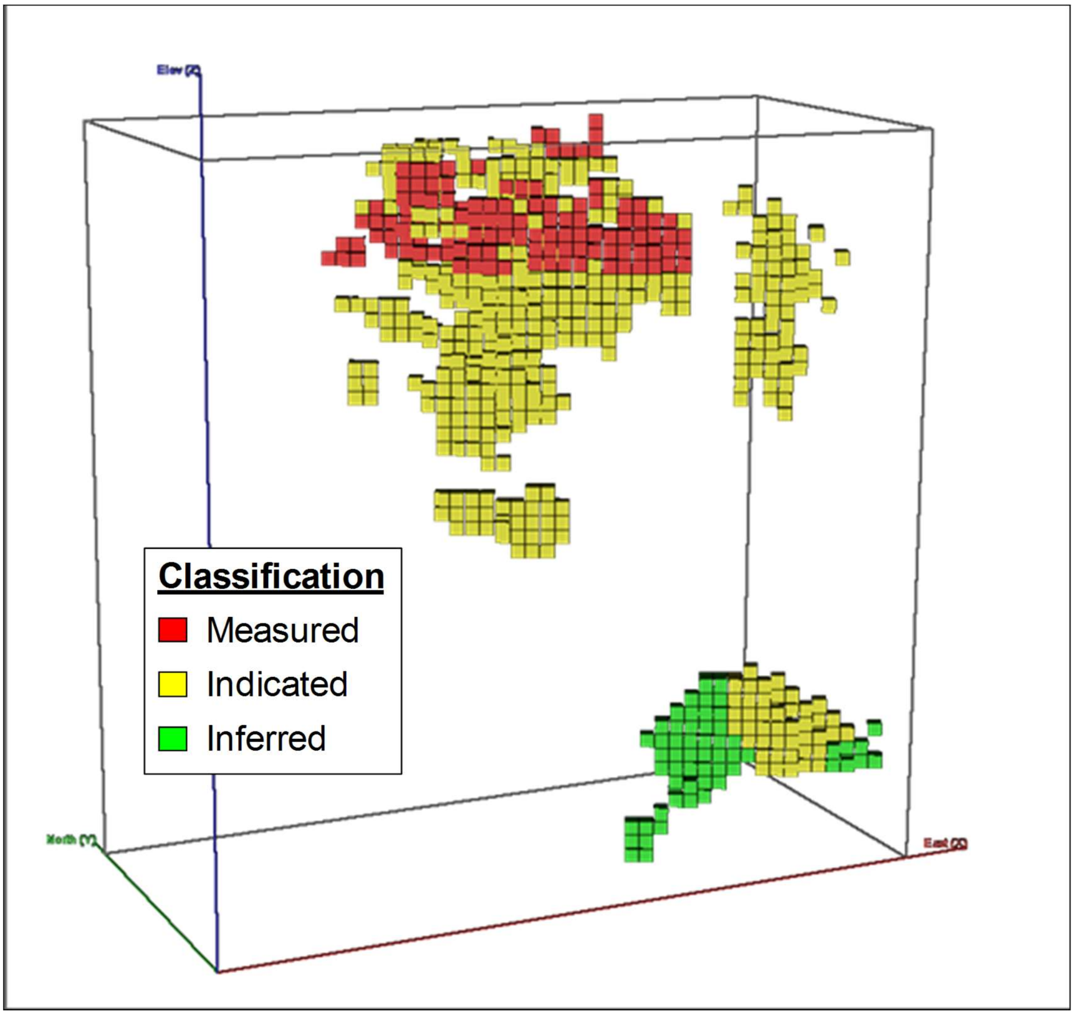


Figure 14.19 – Category distribution above the selected 5.00 g/t Au cut-off grade

15 MINERAL RESERVE ESTIMATES

Mineral reserves were classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves. Mineral reserves for the Fenelon deposit incorporate appropriate mining dilution and mining recovery allowances for the selected mining method.

Mineral reserve calculations estimate the quantity (tonnes) and grade of ore that can be economically mined and processed.

The mineral reserve estimate for the Fenelon deposit is based on the resource block model provided to Wallbridge by InnovExplo, along with information in the InnovExplo report titled "Technical Report and Mineral Resource Estimate for the Fenelon Mine Property", dated August 17, 2016 (Richard and al. 2016).

The conversion of mineral resources into mineral reserves is based on the economic parameters detailed in the following tables. Only mineral resources that have been classified as measured and indicated were used in the economic calculations.

For the reserves and all engineering studies the determination of the price of gold was established by InnovExplo and it was decided to use the September 2016 6-month trailing average of US$1,285. This is included in Table 15.1. This 6-month trailing average exchange rate of 1.31 is also used in the cut-off grade determination.

Table 15.1 – Determination of Gold Price

Month	Price (USD)	CAD/USD	Price (CAD)
Sep-13	$1,349	1.04054762	$1,403
Oct-13	$1,316	1.034335	$1,361
Nov-13	$1,276	1.03673182	$1,323
Dec-13	$1,225	1.049025	$1,285
Jan-14	$1,245	1.063705	$1,324
Feb-14	$1,301	1.09484545	$1,424
Mar-14	$1,336	1.10606316	$ 1,478
Apr-14	$1,299	1.11068571	$1,443
May-14	$1,288	1.09901905	$1,415
Jun-14	$1,279	1.08885238	$1,393
Jul-14	$1,311	1.08258095	$1,419
Aug-14	$1,296	1.07404091	$1,392
Sep-14	$1,239	1.092195	$1,353
Oct-14	$1,222	1.10135238	$1,346
Nov-14	$1,176	1.12166818	$1,319
Dec-14	$1,202	1.13315263	$1,362
Jan-15	$1,252	1.15361429	$1,444
Feb-15	$1,227	1.21231429	$1,488

Month	Price (USD)	CAD/USD	Price (CAD)
Mar-15	$1,179	1.24988421	$1,473
Apr-15	$1,198	1.26120455	$1,511
May-15	$1,199	1.23307619	$1,479
Jun-15	$1,182	1.218475	$1,440
Jul-15	$1,130	1.23576364	$1,396
Aug-15	$1,117	1.28642727	$1,438
Sep-15	$1,125	1.31432	$1,478
Oct-15	$1,159	1.32664762	$1,538
Nov-15	$1,086	1.30730476	$1,419
Dec-15	$1,068	1.32751	$1,418
Jan-16	$1,097	1.37035238	$1,504
Feb-16	$1,200	1.42255	$1,707
Mar-16	$1,246	1.37884	$1,719
Apr-16	$1,242	1.32104091	$1,641
May-16	$ 1,259	1.2817619	$1,614
Jun-16	$1,278	1.29425238	$1,655
Jul-16	$1,337	1.3061	$1,747
Aug-16	$1,348	1.3046	$1,747
3-YR AVERAGE	$1,230	1.1982	$1,469
2-YR AVERAGE	$1,199	1.2606	$1,510
1-YR AVERAGE	$1,204	1.3296	$1,599
LAST 6 MONTHS	$1,285	1.3144	$1,689

15.1 Cut-Off Grade Determination

The operating cost per tonne used in InnovExplo's resource calculation was C$220 per tonne. The cut-off grade was determined using a preliminary pre-feasibility calculation based on quotes from mining contractors. A revised value of C$247 per tonne was used in the cut-off calculation.

The following table lists the parameters used to estimate the cut-off grade required for the mineral reserves estimate.

Table 15.2 – Cut-Off Grade Calculation

Parameter	Currency	Value	Comments
Operating Cost	CAD	$247.00	Per tonne
Net Operating Cost	CAD	$247.00	Per tonne
Gold Price	USD	$1,285	6-month trailing average, early September 2016
Exchange		1.31	6-month trailing average, early September 2016
Gold Price	CAD	$1,689	Per ounce
Gold Ounce (Troy) to Grams		31.1035	
In-Situ Gold Value per Gram	CAD	$54.12	Per gram
Plant Gold Recovery		97%	
Recovered Gold Value per Gram	CAD	$52.50	
Required Plant Feed Grade		4.705	g/t Au
U/G Dilution		15%	Dilution = W/O
Dilution Grade		1.00	g/t Au
Required In-Situ Grade		5.359	g/t Au

For the purpose of determining the mineral reserve estimate, the calculated cut-off grade of 5.359 g/t was rounded down to 5.0 g/t. Taking into consideration the long haulage distance and custom milling, which increase the operating costs, this value is relatively consistent with other mines in the surrounding areas.

15.2 Block Model Validation

The block model was originally created by InnovExplo using Geovia's GEMS software.

After receiving the block model, the percent-based GEMS model was converted into a sub-celled format that is compatible with Datamine Studio 5D Planner software.

Once the block model was converted, the new sub-celled model was validated to verify that tonnes and grades were equivalent to the data from the original percent-based model.

The results of the successful conversion are shown in Table 15.3 below.

Table 15.3 – Difference between imported GEMS data and sub-celled data

Rocktype	Tondiff	Pertondiff	Audiff	Peraudiff
102	63.990177	0.46%	-0.0019732	-0.22%
103	144.85182	0.48%	0.01559514	0.31%
104	32.181984	0.10%	0.00066433	0.01%
105	176.46187	0.23%	-0.0036251	-0.10%
106	324.64843	0.17%	0.00642779	0.25%
109	91.07552	0.19%	-0.0038168	-0.19%
121	140.51224	0.07%	-0.0007276	-0.03%
122	272.49668	0.23%	-0.005093	-0.30%
123	171.10247	0.07%	0.00021786	0.03%
124	-56.50597	-0.14%	-5.509E-05	-0.04%

An additional waste model was provided by InnovExplo, which included some low-grade mineralization located outside the resource zones. The waste data was therefore added to the ore model and used to calculate internal dilution where applicable. This information is particularly useful in areas where two mineralized zones are close to one another and need to be taken as one large stope to prevent any unplanned caving.

15.3 Mineral Reserves Calculation Methodology

Mineral reserves were calculated from the resource block model, using manually generated wireframes (stopes), which were designed based on the established 5.0 g/t cut-off grade.

At the pre-feasibility level, longhole open stoping, uppers open stoping and drift & fill are the three methods selected for the Fenelon mining project because they satisfy the following design criteria:

- Maintain maximum productivity by incorporating bulk-mining methods and operational flexibility, which should result in lower operating costs; and
- Maintain high overall recovery rates.

Additional losses may occur in transit from the stopes to the mill. Hence, a mining recovery factor is applied to the diluted resource to account for these losses. Dilution and recovery factors that were applied to the resource are further discussed in Section 2.4.

Using an external dilution factor of 1.0 g/t throughout, and a recovery factor based on individual stope evaluation, the following table illustrates the calculated reserve categorized by stope type.

The details of the reserve statement can be found in Appendix IV – *Reserve Data Sheet*. Tables 15.4 and 15.5 show a summary of the reserve statement.

Table 15.4 – Reserve Statement

Stope Type	Mined Tonnes	Grade	Mined Grams	External Dilution 1.0 g/t	Mined Diluted Tonnes	Grams of Dilution	Total Grams	Diluted Grade	Recovery	Recovered Tonnes	Grams Recovered	New Ounces
Uppers	24,652	9.93	244,792	15%	28,350	3,698	248,490	8.77	88%	25,018	219,594	7,060
Long Hole	46,521	11.23	522,364	15%	53,499	6,978	529,343	9.89	96%	51,265	509,319	16,375
Top Sill Surface	10,101	10.13	102,357	15%	11,617	1,515	103,872	8.94	96%	11,186	100,336	3,226
Pit Bench	2,222	9.73	21,614	15%	2,555	333	21,947	8.59	97%	2,478	21,289	684
Drift & Fill	2,232	8.76	19,548	5%	2,344	112	19,660	8.39	97%	2,274	19,070	613
Dev. Ore	1,467	8.03	11,783	5%	1,540	73	11,856	7.70	97%	1,494	11,501	370
Broken Ore	3,100	6.14	19,034	0%	3,100	0	19,034	6.14	97%	3,007	18,463	594
Total	**90,295**	**10.43**	**941,492**	**14%**	**103,004**	**12,709**	**954,201**	**9.30**	**94%**	**96,721**	**899,570**	**28,922**

It should be noted that the reserve statement includes broken ore that is currently present underground, as described in the previous resource statement report (InnovExplo, 2016).

Table 15.5 – Reserve Statement by category

Category	Mined Tonnes	Recovered Tonnes	Grams Recovered	New Ounces
Proven	6,321	6,770	62,970	2,025
Probable	83,974	89,951	836,600	26,897
Total	**90,295**	**96,721**	**899,570**	**28,922**

15.4 Mining Dilution and Recovery

Mining dilution percentages were obtained from benchmarking Sudbury's narrow vein mining operations as well as other similar operations in Quebec. Note that the veins are fairly vertical and ground conditions are classified as good; this will be discussed further in section 16.2 of this report. Visual field examination of the core by the QP showed generally good RQD factors. A dilution rate of 15% was applied to both longhole and uppers longhole open stoping, while 5% was applied to drift & fill stoping. Based on block model observations, an average dilution grade of 1.0 g/t was given to the waste rock surrounding the stopes.

The recovery calculations were again made by observing Sudbury and Quebec benchmark operations. Each stope was examined individually and recovery factors from 80% to 97% were applied. The following table lists the typical dilution and recovery factors used to determine the mineral reserve.

Table 15.6 – Dilution and recovery factors

Mining Method	Dilution	Recovery	Comments
Longhole Open Stoping	15%	80-97%	Note veins are mainly perpendicular
Uppers Longhole Open Stoping	15%	80-95%	Some uppers break into open pit and will blast easily
Drift & Fill	5%	97%	Good control on drifting

16 MINING METHODS

16.1 Basis of Design

The proposed mining plan for the Fenelon deposit was prepared using the Measured and Indicated Resources presented in Item 14 – *Mineral Resource Estimates*. These resources were converted to Proven and Probable Reserves based on the parameters described in Item 15 – *Mineral Reserve Estimates*.

The stopes were manually designed using basic parameters discussed by the designer and Wallbridge teams. According to the Wallbridge team, the company intends to maintain a minimum stope width of 2.5 metres, along with a maximum of 12 metres. These parameters correspond closely to the original recommendations of Golder Associates Ltd ("Golder") as stated in their 2004 report Although Golder's stoping design is based on a cut & fill (selective) approach, recent geological information reveals that a bulk mining method is plausible. As a result, the preferred longhole open stoping (bulk) approach should require most stopes to be rock-filled in order to successfully execute the stope sequence and maintain high recovery rates.

It was also agreed with Wallbridge that level spacing be kept around 15 metres, floor to floor. Including the top sill, which adds another 3 metres, the total stope height in most cases remains 18 metres. According to Golder's recommendation, a maximum strike length of approximately 30 m should be in place for a stope 18 metres high. Again, Golder's recommendations are based on an unsupported approach, while the proposed design suggests the use of rock fill. Using Golder's recommendations in combination with rock fill is therefore a cautious approach. A 5-metre rib pillar should be left in place if the strike length of economic ore exceeds 30 metres.

In some areas, the parallel mineralized zones defined in the resource are positioned close to one another. In order to prevent caving of adjacent stopes, a required minimum pillar thickness of 5 metres was settled upon during a discussion with the client. Therefore, the areas that do not meet this parameter were individually assessed. One solution is to combine the two zones together into one large stope. This increases internal dilution. However, if the overall grade remains above 5.0 g/t, this approach is considered economic. If combining two zones into one large stope proves to be uneconomic, the last resort solution is to leave some ore behind. Stope size is mainly dictated by a maximum hydraulic radius factor of 6.0.

The Table 16.1 shows the basis of design derived from existing mine development, as well as geotechnical considerations and current best mining practices.

Table 16.1 – Mine Design Parameters

Subject	Units	Parameters
Mining Methods		Longhole / Uppers / Drift & Fill
Cut-off Grade (CoG)	g/t	5.0
Value of the Ore at CoG	$/t (CAD)	247.00
Stope Mining Rate (Target)	t/d	400
Ramp Development	m	4.5 H x 4.0 W
Level Access	m	4.0 x 4.0

Subject	Units	Parameters
Sill Development	m	3.0 x 3.0
Raise Development	m	2.44 x 2.44
Minimum Stope Width	m	2.5
Maximum Stope Width	m	12.0
Maximum Stope Height	m	18.0
Maximum Stope Strike	m	30.0
Minimum Pillar Required Between Stopes	m	5.0
Production Hole Diameter	in	2-1/4
Lateral Advance Rate	m/d	6.0
Vertical Advance Rate	m/d	2.4

The manually generated stope wireframes were designed using the parameters discussed above. The wireframes were then interrogated against the block model, which reports tonnage and grades within the shapes. The data generated from this interrogation process was then inputted into an Excel spreadsheet. Dilution and mining recovery factors were therefore applied on a stope-by-stope basis.

16.2 Geomechanical Assessment

In January 2004, Golder was retained by International Taurus Resources Inc. to estimate minimum design crown pillar thicknesses. They also listed recommendations for the stope dimensions with consideration that the stopes would not be backfilled. Following this mandate, the draft report titled “Preliminary Assessment of Crown Pillar Stability and Stoping Design for Fenelon Mine, Quebec” was released in April 2004.

Golder’s assumptions were based on underground observations of mineralized zone geometries. Their assessment resulted in the selection of cut & fill or drift & fill as the preferred mining methods.

16.2.1 Rock Mass Classification

Section 5.2.5 in the April 2004 Golder report summarizes how the meta-volcanic and mafic intrusions have generally good rock mass quality. The meta-sediments/argillite and the mineralized mafic volcanic exhibit poor to fair rock quality. The mineralized zones tend to be more fractured than the unmineralized meta-volcanic rocks. As expected, the chlorite schist has very poor to poor rock mass quality.

16.2.2 Stope Dimensions

The design parameters for the mining areas are based on geotechnical recommendations provided by Golder. The stope orientation and dimensions are based on a recommended maximum hydraulic radius of 6 m.

Section 6.2 of the April 2004 Golder report (*Dimension guidelines for stope backs*) states that for practical considerations, and assuming fair quality, stope width should be narrower than 12 metres in order to allow mining of unsupported stopes with reasonable strike length.

Section 6.3 – Based on fair stability conditions, the Golder report indicates the following dimension guidelines for unsupported stope wall lengths:

- No limit at 10 m height;
- Max 34m wall length for 15 m height;
- Max 18m wall length for 25 m height; and
- Max 16m wall length for 30 m height.

16.2.3 Crown Pillar Thickness

In Section 8.1 of the April 2004 Golder report (*Considering stope heights of about 15 meters*), the following recommendations were made:

- A minimum crown pillar thickness of 5m should be implemented;
- For planning purposes, the crown pillar thickness should be set ≥ 1.5 times the stope width, provided that the hydraulic radius for the back of the stope does not exceed 4.5;
- If the stopes are developed using upper blast holes, then a 1 to 2 m buffer zone should be added to the crown pillar thickness to account for blast damage.

16.2.4 Consideration Given to Golder's Report

Golder's study results were used, when practical, for the purpose of this pre-feasibility study. Since 2004, new geological information has been made available, and an updated block model has been created. This new information leads us to believe that a bulk mining method is now a possibility for the Fenelon deposit.

It is recommended that further geomechanical studies be conducted prior to commercial operation since the proposed bulk longhole mining approach differs from Golder's more selective cut & fill mining method.

16.3 Development Method

The following longitudinal looking south shows the development layout schematics of the ramp and levels.

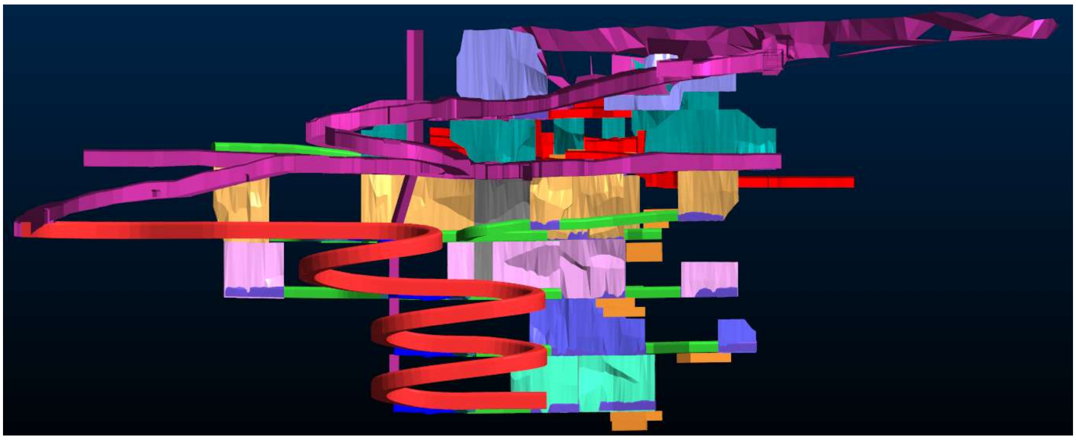


Figure 16.1 – Mine development layout schematic

16.4 Mining Method

The following methods were used to develop the mining reserve.

16.4.1 Longhole Open Stoping Method

Longhole mining consists of drilling a series of vertical holes in the ore from one level to the other. The ore is then blasted in vertical slices. The ore is retrieved from the bottom drift using remote scoop trams. For every horizontal level (approximately 15 m floor to floor), a primary slot (drop raise) is excavated at the extremity of the stope. Blasting of the stope (which can vary between 5 to 30 m in length) is achieved following a longitudinal retreating fashion. All the broken ore is extracted before another blast is taken to ensure a maximum recovery of ore, should any unplanned caving occur. Once the stope is empty, waste rock is used to refill the opened excavation as non-cemented rock fill. In order to blast the second stope on the same level, once again a drop raise is pulled as a primary opening. The second stope is then blasted, mucked and backfilled. The process is repeated until the entire level is mined out.

Figure 16.2 illustrates the concept of the longhole open stoping method.

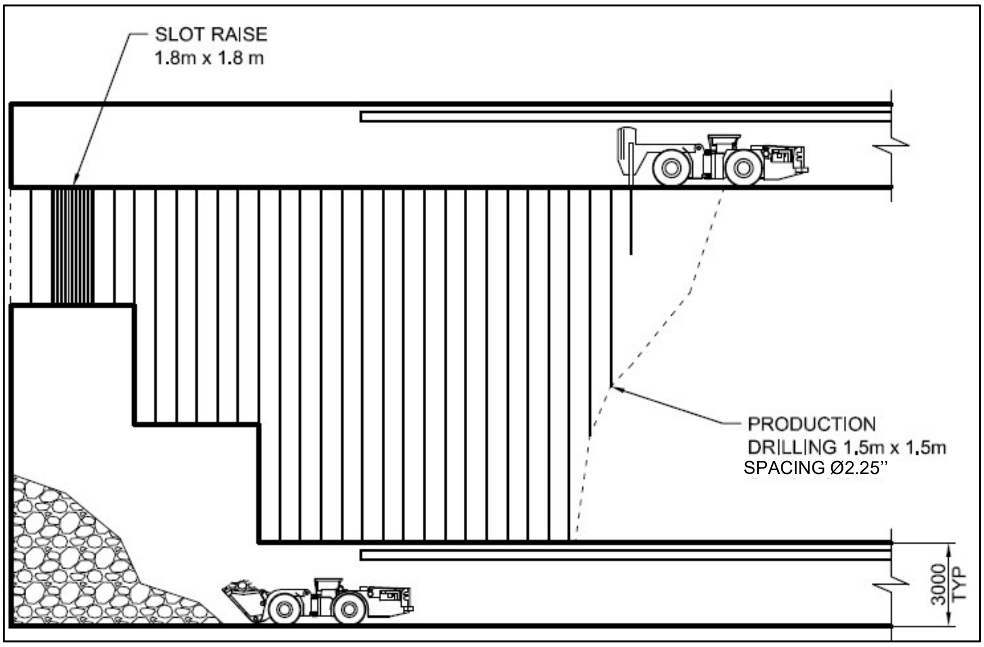


Figure 16.2 – Longhole Open Stoping Method

16.4.2 Uppers Longhole Open Stoping Method

The uppers longhole open stope mining method differs slightly from traditional longhole. Although the recovery is generally not as good, it is a preferred method in areas where the excavation of a top sill proves to be uneconomic. The longhole drilling and loading of the holes takes place at the bottom sill. Once again, a primary slot is excavated at the extremity of the stope and blasting is achieved following a longitudinal retreating fashion. The stope is then mucked out after each blast, in order to ensure maximum recovery of the ore.

Figure 16.3 illustrates the concept of the uppers longhole open stoping method.

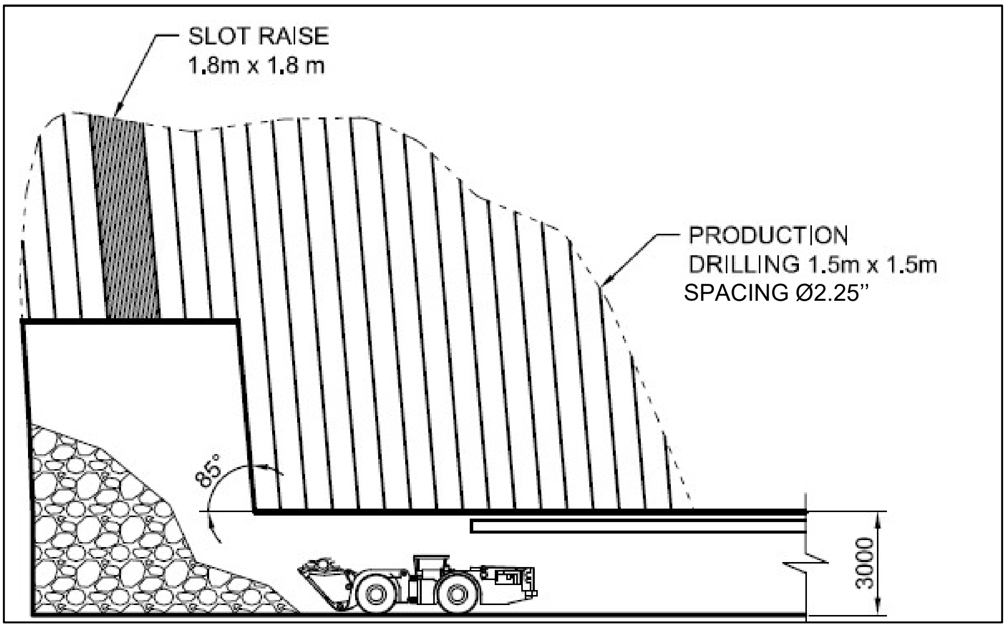


Figure 16.3 – Uppers Longhole Open Stoping Method

16.4.3 Drift and Fill Mining Method

Figure 16.4 illustrates the concept of the drift and fill mining method which should be used in a few locations in the mine when the geometry of the orebody does not permit the longhole or uppers method.

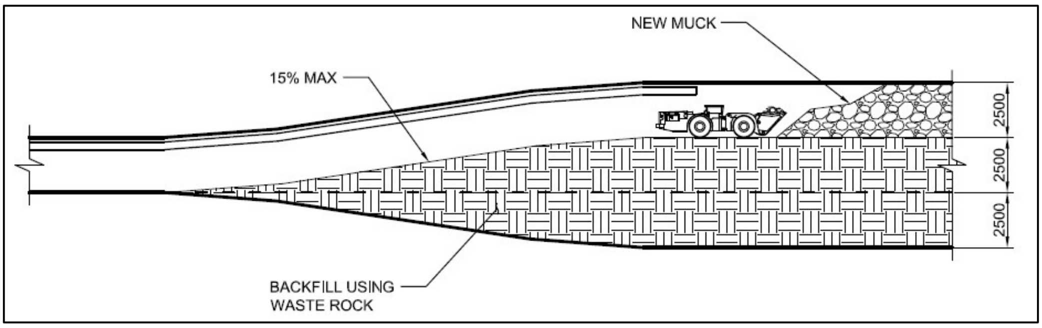


Figure 16.4 – Drift and Fill Mining Method

16.5 Mine Production

The mine production will be carried out by contractors. Scoops will be accessing the production areas from surface via the ramp. To perform an initial evaluation of ventilation requirements, it was assumed that scoops should be used to tram all the material to surface. The necessity and size of trucks should be looked at in more detail at a later time. For instance, the contractors who submitted quotations for the project proposed using different truck sizes. The development rate will influence production outcome as there are multiple mining faces available at a time. A mine schedule was generated based on the design done in Studio 5D Planner, and an average nominal production rate of 400 tpd was determined. Since the custom mill runs optimally in campaigns of at least 10,000 tonnes, an active stockpile should be present on surface. Half loading of roads during the spring thaw may also affect production.

16.6 Mine Operation

The mine will be operated by contractors as a 24/7 two-shift operation. This should be maintained during the life of mine.

16.7 Backfill

At this stage, rock fill should be strategically used in the longhole open stopes. Rock pillars have been included in zones where the strike length of economic ore is greater than 30 metres. These pillars were included in the reserve estimate, therefore they should be mined out initially and filled with cemented backfill. These stopes are identified as stope 48 on level 5150, stope 59 on level 5180 and stope 56 on level 5195. There is an alluvial sand source available about 10 km away from the mine, which may be used to make the cemented sand fill if required.

16.8 Mobile Equipment Fleet

The preliminary mobile equipment fleet should consist of the following:

- 1 – 2 boom jumbo for ramp and level access development;
- 1 – 1 boom jumbo for sill development;
- 1 – 5 yd scoop for ramp and level access mucking;
- 2 – 3 yd scoop for production mucking and backfilling;
- 1 – boom truck for material transportation; and
- 4 – truck personnel carrier.

A more definitive list will be developed during the next project phases in collaboration with the mining contractor selected for the work.

17 RECOVERY METHODS

The results from the testwork done to date in laboratory and during the processing of two bulk samples in commercial plant propose that the preferential way to process the Fenelon ore should be the conventional gold leaching process. For the custom milling processing of this ore, different alternatives exist. The Merrill Crowe, CIL or CIP should normally provide relative equivalent gold recovery. However, no CIL or CIP testwork has been done to date. In the current situation in the Abitibi area, there exist some possibilities for competitive quotations from different processing facilities. The ore is also amenable to flotation, but it could be anticipated that the final gold recovery may be a bit lower than with the cyanide leach. However, this avenue was not optimized at that time. No copper assay in the resources seems to exist so it could be difficult to justify this alternative. For the purpose of this study, it is considered that the ore will be processed at the same facility as the previous two bulk samples. The process facility used was the Camflo Mill located in the Malartic town area and using a Merrill Crowe process.

The Camflo mill circuit could be described as follows:

The crushing circuit begins with a 36 x 48 inch jaw crusher and a primary standard 4½ foot cone crusher in an open circuit. The ore from the primary crusher is stored in a 700-tonne primary coarse ore bin. The secondary crushing is done using a primary standard 4½ foot cone crusher in an open circuit. The product from the primary crusher feeds the secondary 4½ foot short head cone crusher in a closed circuit with a screen with ¾ x ¾ inch openings. The crushing capacity is approximately 125tph. The ¾-inch product is sent to the three ore storage bins of 550, 590 and 680 tonnes, respectively, for a total capacity of 1820 tonnes.

The ore is fed from ore storage to the grinding circuit at a rate of 40-45 tph. The primary grinding is achieved with an 8' x 12' rod mill (450 hp) operated in an open circuit. The secondary grinding is provided by a closed circuit configuration having two ball mills: one 8' x 15' 450 hp and one 9' x 12' 400 hp. Classification is realized with a single cyclone. The underflow is used to feed both ball mills with the overflow as the final grinding product.

The cyclone overflow feeds three similar thickeners of 38 feet in diameter by 14 feet high. The thickener's underflow feeds the leaching circuit. The leaching circuit has 6 leach tanks of 29 feet in diameter by 26 feet high with a capacity of 400 cubic metres each. The circuit is designed with three washing stages. The first washing stage is done after the first three leaching tanks, the second after the fifth leach tank and the final washing at the end of the leaching circuit. The washing is done with two drums filters, 12' in diameter x 16' long, in each washing stage. The leaching retention time is around 45 hours at the nominal capacity. The filtered solution is sent from the filter to the thickener's overflow, which becomes the pregnant solution.

Gold is recovered through a Merrill Crowe process. The process consists of a solution bag clarifier followed by Merrill Crowe Tower and Perrins presses. The gold recovery circuit has four 48" x 48" Perrin presses, two in operation and two on standby. The Perrin presses are cleaned periodically and the gold-bearing precipitate is melted in Wabi furnaces to produce doré bars. The mill has two Wabi furnaces.

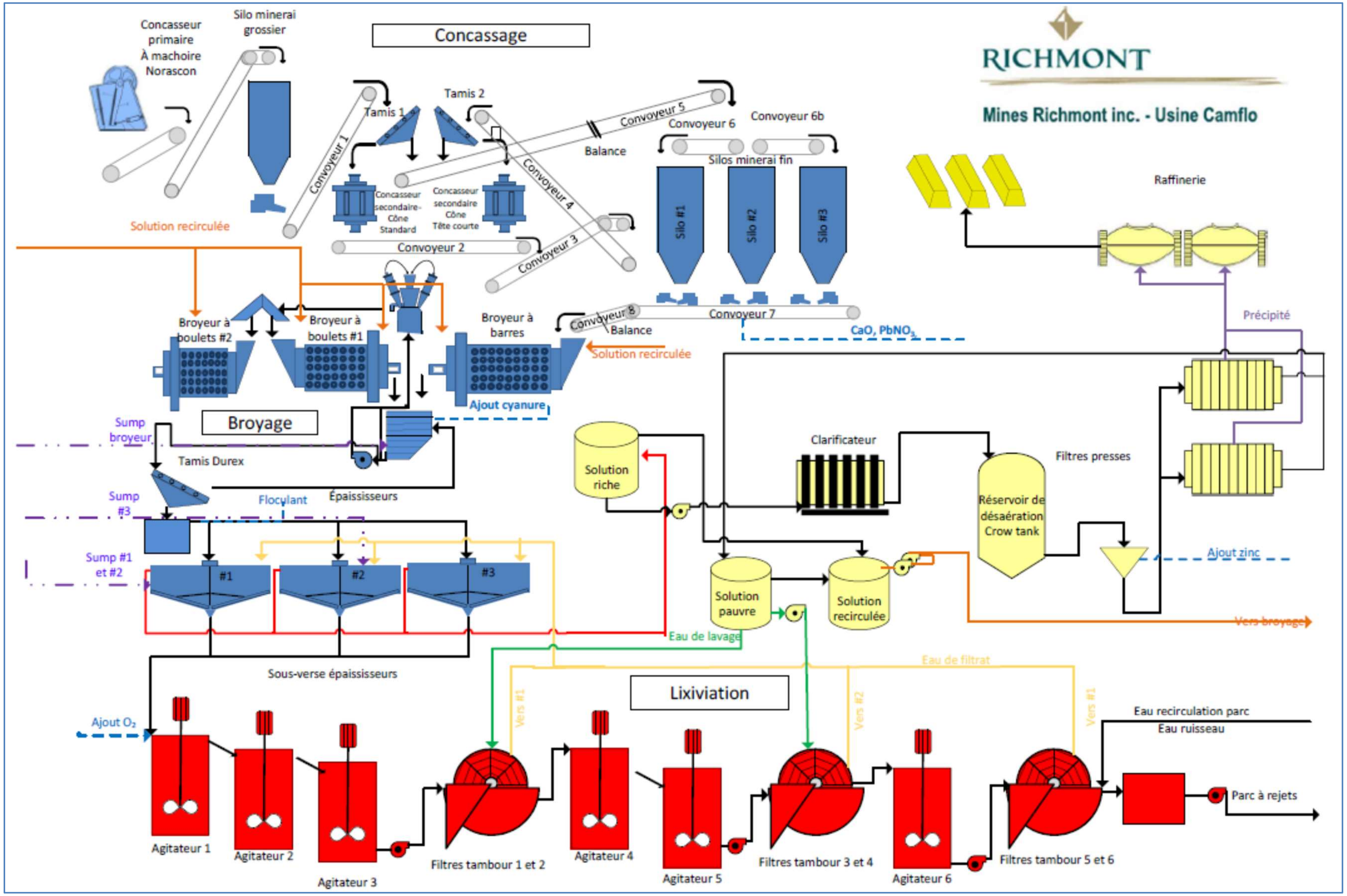


Figure 17.1 – Process flow diagram

*Complimentary schematic provided by Richmont Mines Ltd

18 PROJECT INFRASTRUCTURE

18.1 General Description

The entrance to the mine is shown on the surface general arrangement drawings (1 general drawing and 2 close-up views) in Appendix V. The mine is accessed from La Sarre along a paved road to Val-Paradis, and then by means of a gravel road towards Selbaie Mine. The drive continues north, beyond the Selbaie Mine property. The last 5 kilometers before entering the Fenelon Mine Property is a single lane gravel road. This gravel road also loops back to highway 109 near Joutel, which eventually leads into Amos.

18.2 Access Road Enlargement

Passing bays should be built in the 5-km road between the Balmoral camp and the mine to allow trucks to meet. Radio communication should be used to ensure safety and efficiency on this segment of the road.

The cost for the Selbaie Mine maintenance road is currently covered and shared by users of the road. Wallbridge's projected share of the maintenance cost was estimated and is included in the cost estimate.

18.3 Surface General Arrangement

The surface general arrangement drawing was prepared and shows the location of all surface infrastructure needed for the Fenelon operation, as well as existing infrastructure currently present on the site. The drawing can be found in Appendix V. It is currently expected that the onsite infrastructure will be built mostly by the underground mine contractor, the surface transportation contractor and the camp management contractor.

18.4 Ventilation System 18.4

The basis for the Fenelon ventilation system is the mobile equipment fleet listed in section 16.8 of this report. Regulations from Quebec's Occupational Health and Safety in Mines (chapter 100 and 101) were also considered during the design of the ventilation system.

18.4.1 Air volume

Table 18-1 lists the CFM requirements for Fenelon based on the initial mining fleet equipment list.

Table 18.1 – Mine design parameters

Mobile Equipment	Unit	HP	As per 101-2.a)[1]	HP	144.8 CFM/HP
Truck Personnel Carrier #1	1	100	50%	50	7,240
Truck Personnel Carrier #2	1	100	50%	50	7,240
Truck Personnel Carrier #3	1	100	50%	50	7,240
Truck Personnel Carrier #4	1	100	50%	50	7,240
Sandvik DD311 (1 Boom Jumbo)	1	83	50%	42	6,017
Sandvik DD321 (2 Boom Jumbo)	1	134	50%	67	9,705
Sandvik LH410 (5 yd Scoop)	1	295	100%	295	42,702
Sandvik LH203 #1 (3 yd Scoop)	1	96	50%	48	6,939
Sandvik LH203 #2 (3 yd Scoop)	1	96	50%	48	6,939
Boom Truck	1	138	75%	104	14,987
Total		**1,242**		**803**	**116,250**

Note 1: Règlement sur la santé et la sécurité du travail dans les mines (Québec)

It is estimated there should be about 20 workers underground at any given time. At 529.37 cfm per person, this means an additional 10,600 cfm. Therefore, the total CFM requirement for Fenelon is estimated to be around 127,000 CFM.

18.4.2 Static Pressure

Table 18.2 presents the sizes and lengths of the underground openings used to calculate the static pressure.

Table 18.2 – Static Pressure Calculation

	Length	K (x10-10)	L (ft)	P (ft)	A3	R=KLP/5.2A3
Ramp (15'h x 13'w)	924 m	1.00E-08	3032	56	7414875	4.4031051E-11
Level Access (13'h x 13'w)	203 m	1.00E-08	666	52	7414875	1.3798827E-11
Sill (10.0' x 10.0')	1152 m	1.00E-08	3780	40	1000000	2.9074708E-10
Overall Lateral	2279 m					
Raise (8x8)	48 m	1.20E-08	157	32	262144	4.4364483E-11
						3.9294144E-10
H = RQ2 (for Q=127 000)	6.32 in	≈ 6 in				

18.4.3 Fan Selection

An example manufacturer fan curve was used to determine the fan size and the energy required to provide 127,000 CFM of air flow at 6 inches of static pressure. Figure 18.1 illustrates the selection process.

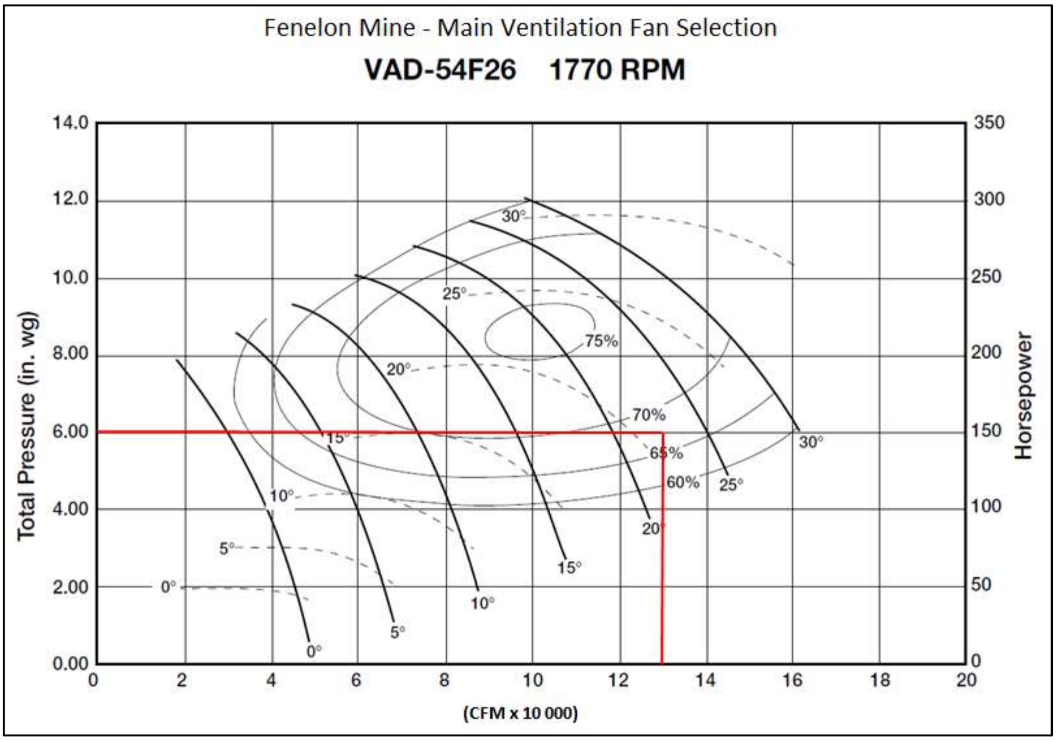


Figure 18.1 – Main ventilation fan selection

The selected unit is a 54-inch axial fan with a 150 hp motor, and blade settings at about 23°. This preliminary fan selection was provided to the mining contractors as backup information for quotation. The selected contractor performed ventilation requirements calculations based on the mining fleet they intend to use and selected a 200 hp fan. Costs for the use of this fan were carried in the project cost estimate.

18.4.4 Ventilation Bulkhead and Man Door

Ventilation bulkheads should be installed on all levels, with the exception of 5225 level. Note that the ventilation raise should not break through on this level.

The bulkheads should be equipped with a 36-inch auxiliary fan driven by a 50 hp motor. This is to provide air to the level and sills. The level accesses should be ventilated using 36-inch diameter flexible ducting. However, the 36-inch ducting should be reduced down to 24 inches once inside the sills to avoid equipment clearance issues. This 24-inch ducting should carry fresh air to the various stoping activities.

An airlock personnel door should also be installed on each bulkhead in order to provide access to the escape way, which is the mine's secondary means of egress. Figure 18.2 illustrates the recommended typical bulkhead design for the Fenelon mine.

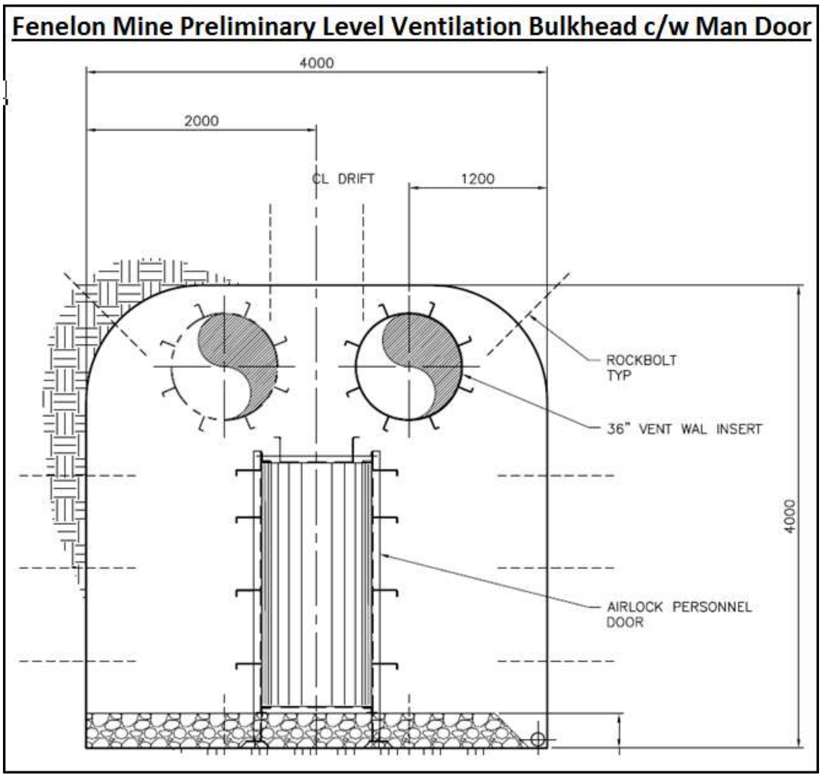


Figure 18.2 –Typical bulkhead design

18.4.5 Ventilation Phases

The mine's ventilation system should be established in three (3) phases, as described below.

18.4.5.1 Phase 1 – Dewatering and Rehabilitation of the Mine

Once the portal is accessible, a 48-inch, 100 hp axial fan should be installed at the entrance. Flexible ventilation ducting should be installed as dewatering and rehabilitation of the existing mine progresses. The exhaust air should naturally flow up the ramp and exit at the portal.

During this time, the main ventilation system can be installed on surface. The basic components of the system consist of the following:

- 54-inch axial fan, 150 hp;
- 127,000 CFM at 6 inches of static pressure;
- 2 x 6 MBTU propane heaters; and
- Stench system.

18.4.5.2 Phase 2 – Activation of the Main Ventilation System

The main ventilation system can be switched on once the escape way from the 5210 level to surface is in place, and the bulkhead inside the 5210 ventilation access drift is constructed. Auxiliary ventilation fan(s) can be connected to the bulkhead, and mine development activities can resume. Again, the exhaust air should naturally flow up the ramp and exit at the portal. The temporary system used during Phase 1 can be decommissioned at this time.

18.4.5.3 Phase 3 – Advancement of the Ventilation System

The ventilation system is extended as development of the mine progresses. As soon as a level ventilation access is excavated, it is important that the Alimak crew setup their nest. Once the raise has broken through and both the escape way and bulkhead are in place, the level can now tap directly into the fresh air system.

Levels are ventilated using 36-inch auxiliary fans connected directly to the ventilation bulkheads. Flexible ventilation ducting is installed as level development progresses in order to bring fresh air to the faces. The diameter of the ducting reduces to 24 inches in the sill areas to accommodate the smaller excavation profile.

Figure 18-3 illustrates the main components of the ventilation system along with the direction of air flowing throughout the mine openings.

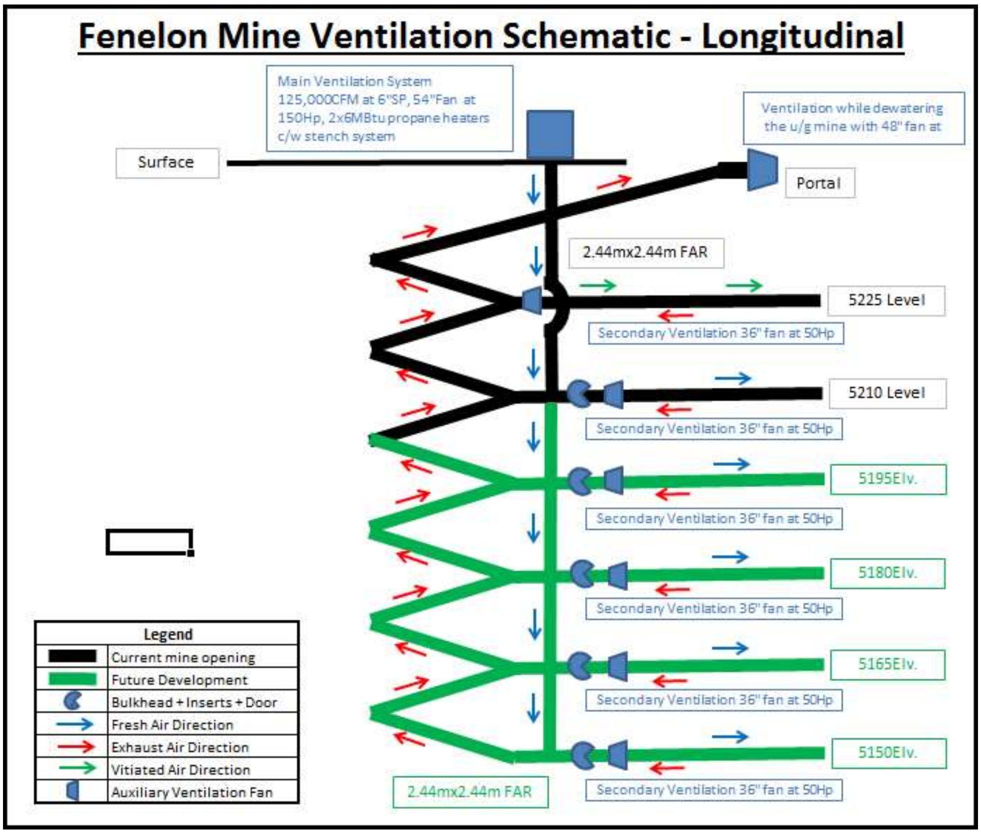


Figure 18.3 – Fenelon mine ventilation longitudinal

18.5 Underground Dewatering System

The dewatering system consists of series of sumps equipped with pumps that essentially move all the dirty mine water to the polishing pond on surface. The design is based on a 200 US gpm pumping capacity. At this point, no hydrogeological modeling was performed as part of the pre-feasibility study.

Dirty water should be moved in stages through a series of 20 hp pumps located on each level. The water should be pumped from the bottom sump on 5150 level, by means of a 4-inch schedule 40 pipe installed in the ramp. Subsequently, the water is dumped into the next sump located on the level above, which in this case would be 5165 level. The underground mine dewatering system is designed to move the water in stages, level by level.

Once the dirty water reaches 5210 level, a 50 hp pump moves the water by means of a 4-inch schedule 40 pipe installed in the ramp. The water moves up towards the portal and exits on surface, where it is dumped into the bottom of the pit. Finally, another 50 hp pump should send the water from the bottom of the pit into polishing pond.

Note that the accumulated slimes in the sumps should be picked up by scoops and brought to surface.

Figure 18.4 illustrates the main components of the underground mine dewatering system.

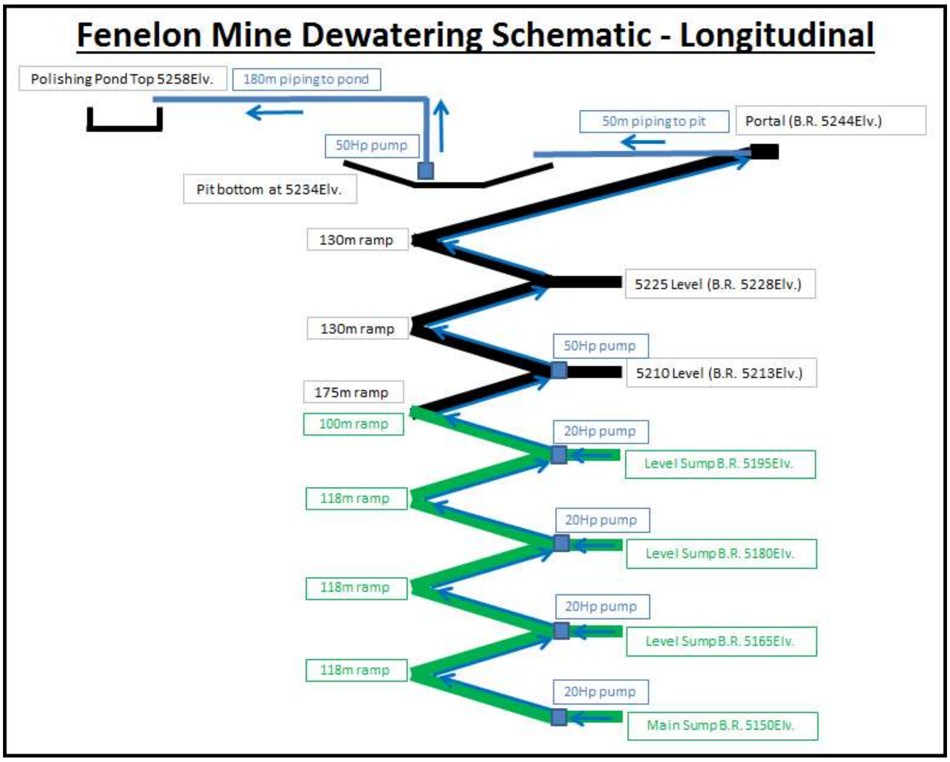


Figure 18.4 – Fenelon mine dewatering schematic

18.6 Power

The contractors should size and install two diesel generator sets on surface to power the mining activities on site. One of the units could power the rooms, office, dry, kitchen, garage and all other surface infrastructure. The other unit could power the mine ventilation and dewatering systems, as well as underground equipment and other related infrastructure.

The preliminary electrical load list and single line diagram are found in Appendix V. Note that all the loads have to be validated to size the generators.

All the electrical installations must be designed and built as per CSA M-421 and C22.10 code.

18.7 Process Water

Process water should be obtained from existing water well located on the site, but it can also be reclaimed from the polishing pond or the surface sump. The clean water is pumped underground for distribution. In order to regulate the pressure, a tote tank should be installed at each level.

18.8 Fuel

Fuel should be trucked in by the contractor and should be used to fill the two diesel generators. A temporary fuel tank placed on a surface pad should also need to be filled by the fuel truck. Note that the temporary fuel tank should be equipped with a Gasboy fleet system dispenser for mobile equipment refueling.

18.9 Pit and Underground Mine Drainage

All underground mine water should report to the dirty water sump located at the bottom of the open pit on surface. The water from the pit bottom should then report to the polishing pond.

18.10 Compressed Air

Compressed air is required for operating underground equipment. It should be provided and installed on surface by the contractor.

18.11 Office Building, Dry, Housing and Cafeteria

The office building, dry, housing and cafeteria should be located at the mine site as shown on the surface drawing. This option is being used as the base case for the pre-feasibility study. For this scenario, a genset should be used to power the camp facility. These facilities should be leased and operated by contractor during the duration of the project.

One other option for these buildings is to lease an existing housing and cafeteria complex already present roughly 5 kilometers from the mine site and increase its capacity. A bus would be required to transport personnel.

18.12 Surface Garage

At this time, there is a large garage on site equipped with a concrete floor. It should be refurbished and made available to the contractor as a maintenance facility. No allowance has been planned for an underground garage at this study level.

18.13 Mobile Refuge and Comfort Station

In accordance with Quebec's *Regulation respecting occupational health and safety in mines* (chapter 126), an underground refuge must be made available to the workers in the Fenelon mine. The mobile refuge should be installed on 5195 level. For more details about this installation, refer to drawing 640914-0000-45DD-0101, located in Appendix V.

Additionally, a comfort station should be installed near the mobile refuge on 5195 level. For more details about this installation, refer to drawing 640914-0000-45DD-0100, located in Appendix V.

18.14 Cap and Powder Magazine

Caps and powder should be delivered underground via the ramp portal for immediate movement to designated underground storage areas. Both cap and powder magazines should be located underground and should be installed according to applicable mining regulations.
Additional details about the cap and powder magazines can be found in drawings 640914-0000-45DD-0102 and 640914-0000-45DD-0103, located in Appendix V.

18.15 Dewatering and Polishing Pond

18.15.1 Mine Dewatering

A mine dewatering system will be required for the two main project phases:

- Initial dewatering of the open pit and existing underground excavations; and
- Ongoing dewatering during development and operation phases.

The current volume of water in the open pit mine has been calculated based on the *Fenelon_topo.dxf* file provided by Wallbridge. Figure 18.5 shows the volume of water in the pit by elevation. During the initial dewatering of the mine, pumps should be placed in the pit where the water should be moved to a polishing pond before being released into the environment. Once the portal can be accessed, dewatering should continue by installing pumps in either the ramp or in the existing ventilation raise, breaking through within the footprint of the flooded pit. The portal elevation is also presented on the Figure 18.5. It suggests that approximately 5,000 cubic metres of storage would be available for the dirty water sump on surface, once the mine has reached the development and operation phases. This water may be used as a source for underground operations. Any excess water should be pumped from this sump into the polishing pond.

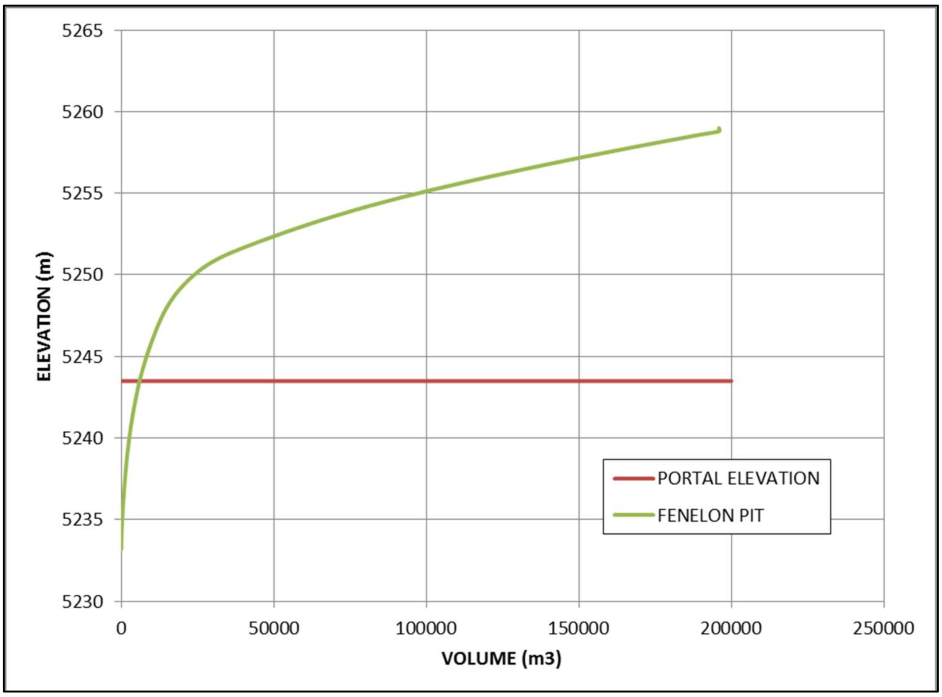


Figure 18.5 – Volume of water in the Fenelon pit.

The estimated volume of water currently stored in the pit is 195,000 cubic metres. In the underground workings, the volume is estimated at approximately 20,000 cubic metres based on the size of available historical development openings, for a total of 215,000 cubic metres of water. It is currently assumed that the peak water flow in the polishing pond should be attained during the initial dewatering phase. The pond is designed with a capacity of 3,800 cubic metres per day, with a safety factor of 18%. At a pumping rate of 3,230 cubic metres per day, the dewatering of the pit would last 60 days. The dewatering of the underground workings would extend the process an extra 26 days, for a total duration of 86 days, excluding rain or snowmelt.

18.15.2 Polishing Pond Arrangement

At this time, a settling pond exists on site and collects the overflow from the open pit prior to releasing it into the environment. To accommodate the development of the site, an expansion of the existing pond to the South is planned to provide additional storage for the dewatering phase. Note that this new arrangement does not take into account some improvements on the flow pattern of the existing settling pond. The current arrangement of the existing settling pond shows potential dead zones and short circuits that can require some additions of baffle for improved performance of the pond. During this study, two possible upgrades to the existing polishing pond were looked at:

- Appendix V (Figure 1A) recommends the relocation of the pond outlet pipe and the addition of a baffle to improve the flow pattern in the pond and benefit as much as possible from a piston flow pattern;
- Appendix V (Figure 1B) recommends the relocation of the pond outlet pipe and no addition of a baffle.

Based on the available information and data, total suspended solids are the only parameter to be improved upon during the water residency in the pond. In fact, available water analyses show that other parameters, such as heavy metals and a pH regulated by Directive 019, are presently met with the current water quality. Therefore, at this time, it is not expected that the project should require a water treatment plant. However, if required, a small plant could be installed after the polishing pond. In the meantime, the polishing pond design is optimized to meet suspended solids criteria at the outlet of the system.

For this phase of the project and given the current performance of the existing pond, the concept in Appendix V (Figure 1B) is considered. Further monitoring of the pond will be required to determine whether the present performance level is being maintained. If not, the concept represented in Appendix V (Figure 1A) should be considered, and the proposed baffle design may be reviewed based on the actual geometry of the existing polishing pond and on the materials available for its construction.

The principles listed in Table 18.3 have been taken into consideration for the extension of the existing pond.

Table 18.3 – Design criteria for the pond extension

Parameters	Unit	Design	Comments
Dewatering Flow	m^3/d	3800 (max)	This includes a safety factor of 18%
Dewatering Duration	days	75 (min)	Minor impact is anticipated if longer period of dewatering is required
Exception Precipitation	mm	0	It is assumed that longer dewatering time should be required if high precipitation occurs during dewatering
Dimensions Length x Width	m	45 x 15	
Length / Width Ratio		> 2	To ensure piston type of flow in the polishing pond
Additional Water Surface Area	m^2	450	
Additional Water Volume	m^3	518	
Type of Material		Compacted Clay	Assumption that the pond should be built with this type of material with slopes of 2H:1V. No slope stability analysis was performed during the prefeasibility study.
Foundation		Clay	Assumption that the in-situ clay layer could be used as an impermeable foundation
Density of Particles to be Captured in the Pond	kg/m^3	2617	
Particle Size	µm	10	Assumption of a silt particle size
Residence Time	hours	12 (min)	Assumption for the polishing pond
Water Depth	m	2	
Freeboard	m	1	
Width at the Bottom of the Pond	m	3 (min)	For maintenance and cleaning purposes

Note that for practical reasons, the existing polishing pond has been reused in order to minimize land area use, schedule and costs. As mentioned above, Appendix V (Figure 1B) presents the proposed polishing pond arrangement.

Also, the polishing pond is not designed to hold any specific recurring event as the dewatering is considered temporary. It is assumed that if such an event occurs during a dewatering process, the dewatering should either be stopped (to avoid the flow of non-compliant water into the environment) and kept in the pit or should be carried on (if the water is still compliant) for a longer dewatering period.

The previously established design parameters are based on limited available information on the existing ground material, topography and water quality. These design parameters are mainly based on the Golder Associates Ltd report and communications with the issuer or its agents. Therefore, it has been assumed that the soil is composed as follows:

- 0.5 meters of overburden;
- 0.5 meters of clay; and
- 7-12 m of silt.

This information is from the report entitled "*Analyse de stabilité des pentes de mort-terrain, projet Fénélon, Québec*", which is a preliminary report released in 2004 by Golder.

For the next phase of the project, it is recommended that more information be obtained on the topography of the site, the parameters of the soil, the existing polishing pond design (and performance) and on the water characteristics. This additional information can be used to update the design parameters presented in this document, if necessary. A hydrology study is also recommended.

18.15.3 Waste Pad

An area to dump waste rock on surface has been selected, and is indicated on drawing 640914-0000-45DD-0001, located in Appendix V. When required, waste rock should be used as rock fill in open stopes. The waste rock is considered to be non-acid generating or metal leaching.

18.15.4 Ore Pad and Crusher

At this time, the ore is considered to be non-acid generating or metal leaching. Therefore, it should be sent through a mobile crusher and stored on surface. Once the ore has been crushed, it should be moved to the mill. The ore pad is designed for temporary storage of up to 30,000 tonnes of ore on surface.

18.15.5 Loading Area

There should be a designated truck loading area on surface in proximity to the surface crusher. A front end loader should be used to load the crushed ore into the haulage trucks. Note that the mine production vehicles and haulage trucks on surface should not interfere with each other. The mill, which was considered for cost estimating and planning, is located 275 kilometres away from the site.

19 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

Markets for doré are readily available and the doré bars produced from Fenelon Mine Property could be sold on the spot market. For the reserves and all engineering studies the determination of the price of gold was established by InnovExplo and it was decided to use the September 2016 6-month trailing average of US$1,285. This is included in Table 15.1. This 6-month trailing average exchange rate of 1.31 is also used in the cut-off grade determination.

19.2 Contracts

No contracts have yet been assigned for the Fenelon Mine Property given the early stage of the project.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Studies

A desktop baseline study has been realized in the context of the environmental impact assessment (EIA) (Gestion Aline Leclerc Inc., 2004a) and the requests for two certificates of authorization (CA) (Roche, 1998, Gestion Aline Leclerc Inc., 2004b). The information provided in the following sections was taken from the documents related to the EIA and the CA requests. The study area for the 2004 EIA comprises the watershed in which the Project is located.

More research information and field surveys may be required to complete or update the environmental baseline for the Project.

The directive from the Evaluating Committee (COMEV) following their analysis of the preliminary information submitted in November 2016 will determine which environmental components will require additional research or field work.

20.1.1 Physical Environment

The following description of the physical environment of the Fenelon Mine Property is based on the following sources of information:

- Public database and documents;
- Information gathered from various governmental agencies as well as other private and public institutions;
- Studies and reports available from Wallbridge;
- Aerial photographs, satellites images, maps and geomatics tools.

The onsite investigation was done on the following:

- Surface water quality;
- Sediment quality.

20.1.1.1 Hydrology

No hydrological study was made available by the previous owners. No hydrological assessment has been performed to date by Wallbridge.

Mine surface water flows into a small intermittent watercourse that drains the bog in which the mining site is located. This watercourse is one of the many tributaries of the Samson River. The Samson River itself is one of the tributaries of the Harricana River flowing northward to James Bay.

The mine site is located within the watershed of the Samson River which flows northwest. It is a small basin of about 90 km², about 70 km² of which is drained upstream of the point of entry of the mining waters into the river system. The watershed located upstream of the point of entry receives about 31,500,000 m³ of water annually.

According to Directive 019 (MDDEP, 2012), the annual and summer low flows (Q_{2-7}, Q_{10-7}, Q_{5-30}) at the point of discharge into the receiving environment must be calculated in order to evaluate the potential impact of the final effluent.

20.1.1.2 Surface Water and Sediment Quality

Surface water quality assessments were completed in 1997 and 2004, upstream and downstream of the final effluent (Roche, 1998 and Gestion Aline Leclerc Inc., 2004c). Three stations were sampled in 1997 and two in 2004. One sample was taken at each station. The majority of the results for the 30 tested elements were all below the MDDELCC surface water quality criteria for both chronic effects and the prevention of aquatic organism contamination (MDDELCC, 2016). However, the 1997 upstream sample yielded a total oil and grease result of 0.49 mg L^{-}1, which is above the chronic effect criteria (0.2 mg L^{-1}). The 1997 sample closest to the discharge point yielded a result of 21 mg L^{-1} for suspended solids, which was more than 5 mg L^{-1} higher than the two other samples (6 and 3 mg L^{-1}). The criterion for chronic effect for suspended solids is defined as an average increase of 5 mg L^{-1} compared to natural or ambient concentrations. However, no effluent or run-off water was discharged before 2001, and no major activities were taking place on the site in 1997. The results may therefore reflect natural concentrations.

Sediment quality assessments were conducted simultaneously with the surface water quality assessments (Roche, 1998 and Gestion Aline Leclerc Inc., 2004c). Particle size and 12 elements were tested on two samples. Sediments found in water bodies in the study area are mostly composed of silt and clay. The 2004 downstream sample yielded results for zinc (88.6 mg L^{-1}) and chrome (37.3 mg L^{-1}), which are above the rare effect concentration criteria (respectively, 80 mg L^{-1} and 25 mg L^{-1}), as well as the three samples from 1997 for chrome (28 mg L^{-1}, 43 mg L^{-1}, 36 mg L^{-1}) (EC & MDDEP, 2007). The 2004 downstream sample and two samples from 1997 also yielded chrome concentrations above the threshold effect level (37.0 mg L^{-1}) (EC & MDDEP, 2007). Since those results are either close to the criteria or encountered in various samples, the results may reflect natural concentrations.

In 2015, the MDDELCC released a new guide for physico-chemical characterization of the initial state of the aquatic environment before the implementation of an industrial project. This guide requests at least 6 to 8 surface water samples, in both exposed and reference areas, to be collected over a one-year period during different hydrological periods, such as the spring freshet and the summer low flow period. For sediment, 3 samples in the exposed area and 1 sample in a reference area should be taken at least once. The MDDELCC may request that the initial surface water and sediment quality initial state characterization be completed for the Fenelon Property.

20.1.1.3 Hydrogeology

No hydrogeology study was made available by the previous site owners. No hydrogeological assessment has been performed to date by Wallbridge.

According to Directive 019 (2012), where mining does not include the development of cyanide, acidogenic, leachable or high-risk tailings impoundment areas, an ore processing plant or a pumping operation exceeding 175,000 m^3 per year, the hydrogeological context may consist of a description of the site based on available geological data. In the case these conditions are found in whole or in part on the site, a hydrogeological study must be carried out. The study area must be within a radius of 1 km around the boundaries of the Project site.

20.1.1.4 Groundwater Quality

No groundwater quality study was made available by the previous site owners. No characterization program has been initiated by Wallbridge at the current state of the Project.

A groundwater monitoring network will have to be installed around risk areas prior to operation. In the case where all the underlying hydrogeological formations are Class III with no hydraulic link, no groundwater monitoring is required. A hydrogeological study is required to determine the classification of the hydrogeological formations.

20.1.1.5 Soil Quality

No soil baseline study was made available by the previous site owners. No soil baseline characterization program has been initiated by Wallbridge at the current state of the Project.

Section 3.3.3.1 of Directive 019 (2012) requests that any new project must carry out an initial site characterization to establish soil quality according to the most recent versions of the *Soil Protection and Rehabilitation of Contaminated Sites Policy*, *Land Protection and Rehabilitation Regulation* and *the Land Characterization Guide* published by the MDDELCC. The Ministry may request that the issuer complete an initial soil quality characterization.

20.1.1.6 Air Quality

No air quality study was made available by the previous site owners. No air quality characterization programs have been initiated by Wallbridge at the current state of the Project.

As of June 2011, the construction or alteration of a stationary source of contamination or an increase in the production of a good or a service is prohibited if it will likely result in an increase in the concentration of a contaminant listed in the *Clean Air Regulation* in excess of the limit for that contaminant.

For some contaminants emitted into the atmosphere, emission modeling may be required to verify compliance with ambient air criteria and to assess their impact. The *Clean Air Regulation* and *the Atmospheric Dispersion Modelling Guide* by the MDDELCC specifies the minimum requirements to complete an air dispersion model.

20.1.1.7 Noise and Vibrations

No ambient noise or vibration studies were made available by the previous site owners. No ambient noise level or vibration baselines have been initiated by Wallbridge at the current state of the Project. Sound levels could be characterized considering the following available data: low intensity land use and limited primarily to mineral exploration, forestry activities and traditional activities carried out by tallymen, no permanent or temporary residence are located near the project site, the exception being the Balmoral exploration camp, 5 kilometres from the Project site.

Instruction Note 98-01 of the MDDELCC indicates that when an area is not zoned for a certain activity within a municipality, real uses determine the zoning category. A noise monitoring program could be required.

According to the Directive 019 (2012), a monitoring network of ground vibration and air pressure must be installed near homes or artesian wells when mining activities take place within 1 kilometre from a point of impact. No homes or artesian wells are within 1 kilometre from the Fenelon Mine Property. Said monitoring program should not be required.

20.1.2 Biological Environment

This section provides information on biological components that may represent a constraint should they be affected by the Project.

The biological environment of the Fenelon Mine Property is based on the following sources of information:

- Public data base and documents;
- Information gathered from various governmental agencies as well as other private and public institutions;
- Studies and reports available from Wallbridge;
- Aerial photographs, satellites images, maps and geomatics tools.

The onsite investigation was done on the following:

- Fish inventory.

20.1.2.1 Vegetation and Wetlands

No exhaustive floral inventory has been made on the sector under study to date. Forest stands in the mine area are mainly mature spruce stands. These stands grow in the area of glacial fluvial deposits, and therefore on xeric to mesic drainage soils. In general, the land is flat and the slopes vary from 0 to 3%. Some spruce-spruce stands and cladonian spruce stands are found. These forest stands are of great importance in the ecology of woodland caribou, as they are the caribou’s main source of food.

The aerial photography and topographic maps indicate that the area consists of numerous peatlands drained by streams. In the best drained areas, trees have established, while in poorly drained areas, shrubby vegetation dominates.

The majority of the site has been developed in the past. The new infrastructure will affect a limited area of vegetation. The former owners committed to preserving the cladonian spruce stand near the mine site as a mitigation measure on woodland caribou. This commitment should be respected by Wallbridge.

20.1.2.2 Wildlife and their Habitats

The *Conservation and Development of Wildlife Act* and the *Regulation respecting Wildlife Habitats* regulates the conservation of wildlife and its habitat and states that no person may, in a wildlife habitat, carry on an activity that may alter any biological, physical or chemical component peculiar to the habitat of the animal or fish concerned.

No exhaustive terrestrial wildlife inventory has been completed to date. Information requests were addressed to the Ministère des Forêts, de la Faune et des Parcs (MFFP) to validate the possible presence of protected wildlife habitat within 5 kilometres of the mine site. The ministry did not report any specific information in this area. However, a herd of woodland caribou is reported further east, near Grasset Lake (Gestion Aline Leclerc Inc., 2004a).

20.1.2.3 Fish and their Habitats

Both federal and provincial governments regulate fish and their habitats. On the provincial level, the MFFP regulates fish habitats under the *Regulation respecting Wildlife Habitats*.

Fisheries and Oceans Canada, by the *Metal Mining effluent Regulations* under the *Fisheries Act*, requires any new project to carry out an Environmental Effect Monitoring program before mining commences. The possible main impact of activities on the Fenelon Mine Property on fish and their habitats will be on the receiving water body at the final discharge point. Compensation measures might be required even if it does not imply habitat destruction or serious harm to fish.

Experimental fisheries were conducted in June 2004 at three sites upstream of the Fenelon site and one downstream. The purpose of these fisheries was to determine the areas that could be selected for a future Environmental Effects Monitoring (EEM) study during the operation phase. Nine species were captured overall. Two species present both upstream and downstream could be used for the monitoring studies.

The MFFP's answers to information requests indicate that 40 species of fish have been found in the area (fisheries areas 16 and 17).

20.1.2.4 Species at Risk and their Habitats

The *Threatened or Vulnerable Species Act* and its regulations apply to threatened or vulnerable wildlife and plant species and their habitats. Information requests were addressed to the *Centre de données sur le patrimoine naturel du Québec* (CDPNQ). After verification of the documents received, it appears that no special-status species are identified near the area where activities or infrastructures are planned.

As mentioned in the wildlife and habitat sections, a herd of woodland caribou is reported further east, near Grasset Lake (Gestion Aline Leclerc Inc., 2004a). The woodland caribou is a vulnerable species.

If it is determined that the Project could impact a habitat where species at risk can potentially exist, field inventories will have to be performed.

20.1.2.5 Protected area

Two protected areas with mining restrictions are present in the Fenelon Mine Property area (GESTIM, 2016): Muskuchii Plain (site #4582) and Harricana River (site # 5956). Two Biological Refuges, 08551R076 (# 22516) and 08562R004 (# 22535), are also present in the area.

The closest abovementioned area, Biological Refuge 08551R076, is 8 kilometres away from the mine site. Another Biological Refuge is located about 10 kilometres from the mine site. As for the two protected areas, they are respectively 9 and 13 kilometres away from the Project.

The mine site is part of a package of mining titles located in an area that has been identified as a priority sector for the creation a large protected area for woodland caribou (Leblond and al., 2015).

20.1.3 Anticipated Environmental Issues

All potential impacts of the Project will be assessed during the EIA that should be conducted by Wallbridge. However, given the Project components described in sections 16 and 18, and based on available environmental data, a preliminary list of the main anticipated issues or impacts has been compiled for the construction, operation and closure/rehabilitation phases. They are presented in Table 20.1.

Project design optimization will aim to reduce the potential impact of environmental issues.

Table 20.1 – Mine site and related infrastructure anticipated issues or impacts

Environmental Components	Anticipated Issues or Impacts	Possible Mitigation Measures
Hydrology	Changes in the local flow regime during both the construction and operation phases.	Temporarily disturbed flows will be progressively re-established after the work to avoid any sudden flow changes.
Hydrogeology	Changes in the local groundwater flow regime during both the construction and operation phases.	During construction, and as needed during the operation, a network of monitoring wells could be established around the new infrastructure to check for changes in water levels and quality.
Surface and Groundwater Quality	Risk of groundwater contamination through accidental spillage of oils, hydrocarbons or any other hazardous substances. Discharge of fine particles and woody debris into the water during construction, operation and rehabilitation phases.	The number of machinery fuelling sites will be minimized to reduce the number of at-risk sites. Any eventual leaks due to faulty valves or human error will be reported to the environmental overseer and, depending on the case, equipment will be repaired. Contaminated water will be immediately pumped and disposed of as per regulations.
Soil and Sediment Quality	Risk of soil contamination through the accidental spillage of oils, hydrocarbons or any other dangerous liquids during all mine life.	The number of machinery fuelling sites will be minimized to reduce the number of at-risk sites. Any eventual leaks due to faulty valves or human error will be reported to the environmental overseer and, depending on the case, equipment will be repaired. Soaked surface soil will be immediately dug up and disposed of as per regulations.

Environmental Components	Anticipated Issues or Impacts	Possible Mitigation Measures
Atmospheric Environment	Emission of dust, GHG and other contaminants into the ambient air generated by the vehicles during construction, operation and closure phases. Effect on air quality due to operations and transportation of ore from the site. Emission of dust from overburden, waste rock and ore piles due to wind erosion.	Use dust suppressor. The machinery used shall meet Environment Canada's emission standards for on-road and off-road vehicles. Minimize machinery idling time. Implement a dust management plan.
Noise and Vibrations	Effect on ambient sound due to construction activity, mining operations and transportation of ore from the site.	Minimize machinery idling time. Noise monitoring program.
Vegetation and Wetlands	Loss of area covered by natural vegetation.	Minimize the Project total footprint.
Wildlife and their Habitats	Loss of habitat during construction phase. Disturbance of wildlife during operation phase.	The construction work will be conducted, if possible, outside the breeding season of the main species present at this latitude.
Fish and their Habitats	Fish habitat alteration due to the mine's final effluent.	Minimize as much as possible encroachment in lakes and watercourses. Reuse of water. Rigorous water management.
Species at Risk and Protected Area	Disturbance of wildlife during operation phase. Loss of area covered by natural vegetation.	The construction work will be conducted, if possible, outside critical biological period. Minimize the Project total footprint.

20.2 Ore, Waste Rock and Water Management

Information on the geology and environmental characterization of the waste rock and ore is available in the following reports:

- *Technical Report and Mineral Resource Estimate for the Fenelon Mine Property*. Technical report according to National Instrument 43-101 and Form 43-101F1 prepared by InnovExplo for Wallbridge Mining Company Ltd, August 2016.
- *Étude d'impact. Projet Minier Fenelon. Territoire conventionné de la Baie James.* Report prepared by International Taurus Inc. and Fairstar Explorations Inc. in January 2004.
- *Projet Minier Fenelon. Document complémentaire à l'étude d'impact du 9 février 2004*. Report prepared by International Taurus Inc. and Fairstar Explorations Inc. in June 2004.

Environmental considerations regarding the waste rock and tailings are outlined below.

20.2.1 Geochemical Characterization of Waste Rock and Ore

The impact study presents the results of acid rock drainage (ARD) potential tests conducted on 7 samples of waste rock and 4 samples of ore, as well as results of metal leaching potential tests conducted on 8 samples of waste rock and 4 samples of ore.

The impact study mentions that the waste rock samples subjected to static acid generation potential tests are non-generating based on the mean net neutralization potential (NNP) values and neutralization potential (NP) / (acidity potential) AP ratio. However, three (3) out of seven (7) samples fall within the uncertainty area for which no decision can be made on the potential acid generation (PAG) of the samples. In addition, these samples all have sulfur percentages (S%) greater than 0.9%. The results obtained from the ore samples fall within the uncertainty zone (4 out of 4 samples).

The impact study mentions that the leaching tests (Toxicity Characteristic Leaching Procedure: TCLP) carried out on the waste rock samples indicate low-leachability in Cd and Ba. However, since that report was completed, updates have been made to both the MDDELCC's Directive 019 standards and the *Resurgence dans les eaux de surfaces* (RES) criteria. According to current regulations, the leaching results would characterize the waste rock as leachable in Ba, Cd, Cu, Mn, Ni and Zn, and the ore as leachable in Cd, Cu and Mn. However, no analysis was conducted to determine whether the waste rock and ore would be classified as high-risk as defined in Directive 019.

According to the geological data, four (4) distinct lithologies are observed in the deposit, namely a mafic intrusive, a felsic intrusive, metasediments and a quartz-sericite-chlorite shale. The ore would be associated with the silicification of the geological units and the most abundant sulphurous minerals would be pyrrhotite (trace to 30%) and pyrite. The presence of chalcopyrite is also observed. Since pyrrhotite is the most reactive sulphide capable of causing acid mine drainage (AMD), the following actions are recommended to enhance the waste rock and ore characterization:

- Select new samples in the 4 lithological units (5 to 10 samples per lithology) with the help of project geologists;
- Carry out analyses of available metals (scan) since these have not been carried out in the context of the impact study;
- Conduct PAG static testing (MABA method) on selected samples;
- Carry out leach tests (TCLP, SPLP, CTEU-9) on selected samples;
- Verify if the waste rock and ore are considered high-risk according to Directive 019;
- Carry out all analyses to determine whether the waste rock can be reused for construction work (access roads, concrete, etc.);
- Perform kinetic tests (wet cell or column test) on uncertain PAG samples.

Ore and waste rock management should be re-assessed at the next stage of engineering, once the geochemical characterization results of these materials are better known.

20.2.2 Ore Management

The ore will be sent through a mobile crusher located next to the ore stockpile at surface. The ore pad is designed for temporary storage of up to 30,000 tonnes of ore. No ore will remain in the stockpile at the end of operations. At closure, the ore pad will be levelled and covered by organic matter from the overburden pile. The surface will then be revegetated.

20.2.3 Waste Rock Management

A limited amount of waste rock, approximately 60,000 tonnes, will be generated during the mine development. Due to the production and the mining sequence, a surface waste rock pile is required. However, at closure, no waste rock should be left on the waste rock pad as the waste rock will be used as underground filling and as construction material in the development of the ore pad expansion. At this point it is assumed that the waste rock material is not potentially acid generating and not leachable. Waste rock management should be re-assessed at the next stage of engineering. At the end of the mining operation, the waste rock pad will be levelled and covered by organic matter from the overburden pile. The surface will then be revegetated.

20.2.4 Water Management

Underground water management is defined by the two main project phases: the initial dewatering of the open pit and existing underground excavation, and the ongoing dewatering during the development and operation phases.

During the pre-production phase, the water contained in the pit, estimated at 195,000 m^3, will be pumped to the existing polishing pond before being released into the environment. Once the pit will be dewatered, water residing inside the former underground mine openings, estimated at approximately 20,000 m^3, will also be pumped to the polishing pond.

Water in the polishing pond flows through a decant tower located in the existing pond and is released in the environment through an 8-inch pipe. The final effluent will be sampled according to the frequency established in Directive 019 and will comply with the standards for all parameters of the regular monitoring. Wallbridge commits itself to satisfy Directive 019 and to put in place corrective measures if water quality does not meet these standards. Prior development activities were monitored for water quality in the effluent. No particular problems were recorded at the time of the 2004 bulk sampling for the monitored parameters. The mean pH was between 6.5 and 9.5, and the maximum annual concentrations were as follows: arsenic 0.025 mg L^{-1}, copper 0.01 mg L^{-1}, iron 1.268 mg L^{-1}, nickel 0.033 mg L^{-1}, lead 0.02 mg L^{-1}, zinc 0.066 mg L^{-1} and suspended solids 15.8 mg L^{-1} (MDDEP, 2006). However, on two occasions, the former site owners received an environmental infraction from the Ministry for high iron and suspended solid concentrations. Those parameters will have to be closely monitored.

As described in section 18, it is important to note that the polishing ponds are not designed to hold any specific recurring event as they are designed for temporary dewatering purposes. It is assumed that if such event occurs, the dewatering process will either be stopped (to avoid the flow of non-compliant water into the environment) and water will be hold in the pit or dewatering will be carried on for a longer period (if water is still compliant).

Typically, run-off water from the waste rock, overburden and ore piles is collected and its quality monitored, and if required, the water is treated before its release to the environment. At the next phase of engineering, a surface water management plan will need to be designed.

Run-off water from the waste rock, overburden and ore piles will either be collected by a system of ditches and conveyed to the open pit or gravitationally directed towards the pit. Sizing of the required collection basins and water management infrastructure will need to be revised at the next stage of engineering.

A clean water diversion ditch is actually in place around the open pit preventing clean run-off water to enter the site's drainage infrastructure.

20.3 Regulatory Context

20.3.1 Environmental Impact Assessment Procedure

20.3.1.1 Provincial Procedure

The environmental impact assessment (EIA) procedure in the province of Québec is based on two regimes: Southern and Northern Québec. By virtue of its location, the Fenelon Mine Property falls under the Northern Québec regime. In accordance with the James Bay and Northern Québec Agreement and the Northeastern Québec Agreement, Chapter II of the *Environment Quality Act* (EQA) contains specific provisions applicable to the Baie-James and Nord-du-Québec administrative regions. The particular environmental assessment procedures for these northern regions stand apart due to, among other things, the active participation of Cree communities.

The mine infrastructure is located south of the 55th parallel and therefore falls under sections 153 to 167 of the EQA. It requires any person or group to follow the EIA procedure before undertaking a project targeted by Schedule A of the Act. Schedule A paragraph (a) stipulates that all mining development, including additions, alterations or modifications to an existing mining development project, is subject to the provincial procedure. The Fenelon Mine Property should therefore be the subject of an environmental and social impact assessment (ESIA).

The *Regulation respecting the environmental and social impact assessment and review procedure applicable to the territory of James Bay and Northern Québec* provides details on the information that should be included within the environmental impact study. The Evaluating Committee (COMEV) is the agency responsible for assessing and drawing up guidelines for the impact study. The Review Committee (COMEX) is the agency responsible for reviewing projects and the public consultation. The COMEX recommends whether the project will be authorized or refused, and if appropriate, specifies conditions for its implementation. The MDDELCC takes into consideration the decision of the COMEX in determining whether to approve the project and issue a certificate of authorization.

The EIA procedure follows these five general steps:

Preliminary information: A notice of intent and preliminary information is provided to the MDDELCC. This information includes the purpose, nature and scope of the project, as well as possible variants in terms of location or layout.

Evaluation: The MDDELCC sends the notice of intent to the COMEV. They formulate guidelines outlining the extent of the impact study to be prepared by the proponent. These guidelines are submitted to the MDDELCC, who transmits them to the proponent, with or without changes.

Environmental Impact Assessment: The proponent conducts the environmental and social impact study, which must take into account the directive issued by the MDDELCC and the *Regulation respecting the environmental and social impact assessment and review procedure applicable to the territory of James Bay and Northern Québec.*

Review: The proponent submits the EIA to the MDDELCC who then sends it to the review committee (COMEX). The Native administrations and the public can make representations to the committee, which may also hold public hearings or any other type of consultation. The COMEX recommends whether to reject or authorize the development project and, if so, under what conditions. It must then define the changes or additional measures that it considers appropriate.

Decision and Authorization: Taking into account the COMEX recommendations, the MDDELCC grants or refuses authorization for the project. This authorization does not exempt the proponent from obtaining sector authorizations that may be required by other law or regulation.

20.3.1.2 Federal Procedure

The federal government requires an ESIA for projects covered under the *Canadian Environmental Assessment Act, 2012* (CEAA 2012).

The CEAA 2012 applies to projects described in the *Regulations Designating Physical Activities*. The project on the Fenelon Mine Property is not submitted to the federal EIA process according to section 16(c) since it does not involve the construction, operation (and, eventually, the decommissioning and closure) of a new gold mine, other than a placer mine, with an ore production capacity of 600 tpd or more.

20.3.2 Laws and Regulations

Following the EIA procedure and the release of the provincial authorization, the Project will require a number of approvals, permits and authorizations prior to initiation and throughout all stages of the project. In addition, Wallbridge will be required to comply with any other terms and conditions associated with the authorization issued by the provincial and federal regulators.

The most significant laws, regulations and directives among the legislation and government directives to be considered and respected are presented below. Their applicability will have to be reviewed as the Project components are defined.

Provincial Jurisdiction

<u>Mining Act</u> (c. M-13.1)

- Regulation respecting mineral substances other than petroleum, natural gas and brine (M-13.1, r. 2)

<u>Environmental Quality Act</u> (c. Q-2)

- Regulation respecting the application of section 32 of the Environment Quality Act (Q-2, r. 2)
- Regulation respecting the application of the Environment Quality Act (Q-2, r. 3)
- Clean Air Regulation (Q-2, r. 4.1)

- Regulation respecting industrial depollution attestations (Q-2, r. 5)
- Regulation respecting pits and quarries (Q-2, r. 7)
- Regulation respecting the declaration of water withdrawals (Q-2, r. 14)
- Regulation respecting mandatory reporting of certain emissions of contaminants into the atmosphere (Q-2, r. 15)
- Regulation respecting halocarbons (Q-2, r. 29)
- Regulation respecting hazardous materials (Q-2, r. 32)
- Protection Policy for Lakeshores, Riverbanks, Littoral Zones and Floodplains (Q-2, r. 35)
- Water Withdrawal and Protection Regulation (Q-2, r. 35.2)
- Land Protection and Rehabilitation Regulation (Q-2, r. 37)
- Regulation respecting the charges payable for the use of water (Q-2, r. 42.1)
- Directive 019 sur l'industrie minière (2012)
- Protection and Rehabilitation of Contaminated Sites Policy (1998)

<u>Threatened or Vulnerable Species Act</u> (c. E-12.01)
- Regulation respecting threatened or vulnerable wildlife species and their habitats (E-12.01,r.2)
- Regulation respecting threatened or vulnerable plant species and their habitats (E-12.01,r.3)

<u>Compensation Measures for the Carrying out of Projects Affecting Wetlands or Bodies of Water Act</u> (M-11.4)

<u>Watercourses Act</u> (c. R-13)
- Regulation respecting the water property in the domain of the State (R-13, r. 1)

<u>Sustainable Forest Development Act</u> (c. A-18.1)
- Regulation respecting standards of forest management for forests in the domain of the State (A-18.1, r. 7)

<u>Conservation and Development of Wildlife Act</u> (c. C-61.1)
- Regulation respecting wildlife habitats (C-61.1, r. 18)

<u>Lands in the Domain of the State Act</u> (c. T-8.1)

<u>Building Act</u> (c. B-1.1)
- Safety Code (B-1.1, r. 3)
- Construction Code (B-1.1, r. 2)

<u>Explosives Act (</u>c. E-22)
- Regulation under the Act respecting explosives (E-22, r. 1)

<u>Cultural Heritage Act</u> (c. P-9.002)

<u>Occupational Health and Safety Act</u> (c. S-2.1)
- Regulation respecting occupational health and safety in mines (S-2.1, r. 14)

Highway Safety Code (c. C-24.2)

- Transportation of Dangerous Substances Regulation (C-24.2, r. 43)

Federal Jurisdiction

Fisheries Act (R.S.C., 1985, c. F-14)

- Metal Mining Effluent Regulations (SOR/2002-222)

Canadian Environmental Protection Act (S.C. 1999, c. 33)

- PCB Regulations (SOR/2008-273)
- Environmental Emergency Regulations (SOR/2003-307)
- Federal Halocarbon Regulations (SOR/2003-289)
- National Pollutant Release Inventory

Species at Risk Act (S.C. 2002, c. 29)

Canada Wildlife Act (R.S.C., 1985, c. W-9)

- Wildlife Area Regulations (C.R.C., c. 1609)

Migratory Birds Convention Act, 1994 (S.C. 1994, c. 22)

- Migratory Birds Regulations (C.R.C., c. 1035)

Nuclear Safety and Control Act (S.C. 1997, c. 9)

- General Nuclear Safety and Control Regulations (SOR/2000-202)
- Nuclear Substances and Radiation Devices Regulations (SOR/2000-207)

Hazardous Products Act (R.S.C., 1985, c. H-3)

Explosives Act (R.S.C., 1985, c. E-17)

Transportation of Dangerous Goods Act (1992)

- Transportation of Dangerous Goods Regulations (SOR/2001-286)

20.3.3 Environmental Permitting Schedule

A general environmental assessment schedule is presented in Table 20-2. Steps highlighted in grey represent a statutory analysis delay. Based on the experience of previous projects in the Northern Quebec regime, it could take 365 to 550 days (12 to 18 months), or more, to receive a provincial authorization. However, since the information from the previous EIA is available and few new areas are affected, the completion time for the item *Realization and submission of the impact assessment statement (including baseline studies)* could be faster, thereby lowering the overall authorization timeline for the project on the Fenelon Mine Property.

Table 20-3 provides a list of required or potentially required permits and authorizations based on the known components of the project on the Fenelon Mining Property and the typical components of a mining project. The usual timeframe required for each authorization to be issued is generally between 1 to 3 months. The table will have to be reviewed at all stages of the project.

Table 20.2 – General provincial environmental assessment schedule

Steps	Duration (days[a])
Notice of intent	-
Guidelines transmission	30[b]
Realization and submission of the impact assessment statement (including baseline studies)	315
COMEX recommendations	45[b]
Representations: Native people and the public	c
Ministry compliance analysis and transmission of questions	45
Submission of answers to questions	30[d]
Issuance of the government certificate of authorization	60
Total	525

Notes: ***a) Calendar days***
b) The time fixed may be extended by the Ministry.
c) There is no determined period of time for information and consultation under the JBNQA.
d) The time fixed depends on the extent of the requested supplementary information.

Table 20.3 – Preliminary and non-exhaustive list of required permits and authorization

Project Components	Request	Government Authority	Legal References
Provincial requirements			
Mine dewatering	CA	MDDELCC	Section 22 of the EQA
Other deposit appraisal activities (i.e., bulk sample)	CA	MDDELCC	Section 22 of the EQA
Mining operation	CA Depollution Attestation	MDDELCC MDDELCC	Section 22 of the EQA Section 31.10 of the EQA
Location of mill, concentration plant and tailings site	Authorization Lease	MERN MERN	Section 240 and 241 of the Mining Act Section 47 of the Lands in the Domain of the State Act (if outside mining lease)
Rehabilitation and restoration plan	Approval	MERN	Section 232.2 of the Mining Act
Pits and quarries operation or crushing activities	CA	MDDELCC	Section 22 of the EQA
Equipment to prevent or reduce the issuance of contaminants into the atmosphere	Authorization	MDDELCC	Section 48 of the EQA
Surface mineral substances extraction	Lease	MERN	Section 140 of the Mining Act
Oil-water separators	CA	MDDELCC	Section 22 of the EQA
Effluent or water treatment facilities	CA	MDDELCC	Section 22 of the EQA
Groundwater intake	Authorization	MDDELCC	Section 31.75 of the EQA
Waterworks, sewers and waste water treatment	Authorization	MDDELCC	Section 32 of the EQA
Clearing	Permit	MFFP	Section 73 of the Sustainable Forest Development Act
Infrastructure implantation on public land (if outside mining lease)	Lease	MERN	Section 47 of the Lands in the Domain of the State Act
High-risk petroleum equipment	Permit	RBQ	Section 120 of the Safety Code
Explosives possession, magazine and transportation	Permit	SQ	Section 2 of the Explosives Act
Explosives magazines site	Lease	MERN	Section 47 of the Lands in the Domain of the State Act
Federal requirements			
Explosives manufacturing plant and magazine Explosive transportation	Licence Permit	MNR	Section 7 of the Explosives Act
Use of radiation devices	Permit	CNSC	Section 3 of the Nuclear Substances and Radiation Devices Regulations
Hazardous substances management set out in column 1 of Schedule 1	Notice and emergency plan	Environment Canada	Sections 3 and 4 of the Environmental Emergency Regulations
Municipal requirements			
CA submitted to the MDDELCC, sections 22 or 32 of the EQA	Certificate of compliance	Eeyou Istchee James Bay Regional Government	
Building construction or modification Building repair, renovation or demolition	Construction permits	Eeyou Istchee James Bay Regional Government	

20.4 Social Considerations

20.4.1 Consultation Activities

In the context of the Fenelon Project initiated in 2004, International Taurus Inc. and Fairstar Explorations carried out consultation meetings with the Cree First Nations of Waskaganish, Washaw Sibi, and possibly Waswanipi, as well as the Algonquin First Nation of Abitibiwinni in order to explain the different stages of the mine. It was agreed that communication would be established between the organizations representing the communities concerned and, in more broadly, with the representatives of regional organizations so they would be aware of the project and understand its impacts (Gestion Aline Leclerc Inc., 2004a).

A stakeholder consultation plan will be put in place with Wallbridge's relaunch of the a project on the Fenelon Mine Property to ensure regular communication with the communities. The consultation activities will be aimed at informing and consulting people living in the territory throughout the process, from project planning to the end of the exploitation of the mine. The consultation plan will be developed to assess the perceptions of the Project by the Cree, Algonquin and Jamesian communities and to identify appropriate mitigation measures.

To this end, letters sent in November 2016 to Algonquin and Cree communities announced that Wallbridge has acquired the Fenelon Mine Property and has recently undertaken the environmental impact assessment procedure. Community representatives were invited to meet with the issuer in order to discuss various topics relating to the Project. Such meetings would also determine how best to hold the consultations with community members who could be affected by the Project, or if other First Nations communities should be included.

At meetings held in the past, there was much talk about employability, not only in terms of onsite jobs and contractors, but also the hiring of specialized firms with which the Crees have partnerships. The environment was also mentioned as a priority issue.

Additionally, in December 2016, an agreement was concluded by the Quebec Government and the Abitibiwinni Nation regarding consultation and accommodation with the mining sector. The objective of this agreement was to clarify the consultation process and determine a territory of application. The Fenelon Mine Property would be affected by this agreement, which will be signed and made public in the near future.

20.4.2 Social Components and Related Requirements

20.4.2.1 Territorial regime and governance

The Fenelon Mine Property is located in the territory of the Eeyou Istchee James Bay Regional Government (EIJBRG) (Government of Québec, 2016). Founded in 2014, the EIJBRG brings together the nine Cree communities of Nord-du-Québec, four Jamesian municipalities (Chibougamau, Chapais, Lebel-sur-Quévillon and Matagami), as well as the three Jamesian communities of Valcanton, Radisson and Villebois (EIJBRG, 2016).

According to information from the MERN (MRNF, 2007), the Project is located on trapline #13, which is listed as an Algonquin trap territory linked to the Pikogan community. This trapline is also overlapped in its northern portion by traplines A-4 and N-8, which are managed by Cree communities according to the Cree Trappers Association. The following Aboriginal communities are involved in the Project: the Pikogan Algonquins (Abitibiwinni First Nation Council), the Waskaganish, Waswanipi and Washaw Sibi Crees. The latter, whose members are not yet grouped in a Cree community, was recognized in 2003 as the 10th Cree Nation by the Grand Council of the Cree. Their status is not officially recognized by the government (Washaw Sibi Eeyou, n.d., GCC, n.d.).

Each community is administered by a band council. Pikogan is a member of the Algonquin Anishinabeg Nation Tribal Council while all Cree communities are headed by two regional political and administrative organizations, the Grand Council of the Crees (GCC) and the Cree Nation Government (CNG). The GCC is responsible for the application of the James Bay Northern Quebec Agreement (JBNQA) to which it is a signatory. The CNG represents the Cree when the JBNQA requires it in areas such as the environment, hunting, fishing and trapping, and economic and community development (GCC, n.d., Hydro-Québec, 2004).

Nord-du-Québec is governed by the JBNQA and by the Paix des Braves, an agreement concerning a new relationship between the Government of Quebec and the Cree of Quebec. This last agreement guarantees the participation of the Cree in the forestry, mining and hydroelectric development of the territory and led to the Agreement on Governance in the territory of Eeyou Istchee James Bay (EIJB).

The territorial regime introduced by the JBNQA is a decisive element in the use of the territory. It provides for the division of the territory into Category I, II and III lands. The Project is located on Category III land. On these lands, the Cree have the exclusive right to trap fur animals. They may establish any camp for hunting, fishing and trapping and, in this case, a title of the Government of Quebec is not required. Moreover, the Crees do not need a licence to practice these activities and there is no limit on the number of catches. In these territories, hunting and fishing are permitted for both aboriginal and non-aboriginal people.

Responsibility for the development and management of Category III land resources is shared between two principal representatives: the MERN and the EIJBRG. Development agencies are also involved in regional planning, including the Administration Régionale Baie-James and the James Bay Regional Commission for Natural Resources and the Territory.

The Public Land Use Plan (*Plan d'affectation du territoire public* – PATP) and the Regional Public Land Development Plan (*Plan régional de développement du territoire public* – PRDTP) are two of MERN's main public management and land use planning tools. According to information obtained from the MERN, there are no PATP and PRDTP currently available for the Nord-du-Québec region.

20.4.2.2 Population

In the EIJB territory, all nine Cree communities numbered 17,468 in 2015 (ISQ, 2016). The Jamesian population was nearly 14,100. The Waskaganish Cree community, which is more than 160 kilometres north of the Project site, is home to 2,380 people, while Waswanipi, more than 190 kilometres east of the proposed mine site, has 1,935 people. The community of Pikogan is located close to the city of Amos, at more than 160 kilometres southeast of the Fenelon Mine Property. The Council of the Abitibiwinni First Nation counts 1,030 members, nearly half of them living outside the community (Dialog, 2012). Finally, the Jamesian town of Matagami, with its 1,500 inhabitants, is located 75 kilometres northwest of the Project site.

20.4.2.3 Land Use

The Fenelon Mine Property appears to be used very little by the neighboring communities because the ecological characteristics of the territory limit the potential for use and development. According to available information (MERN, 2014), there are seven (7) hunting camps and one (1) cottage within a radius of 10 kilometres, only one of which lies within a radius of 5 kilometres from the site. According to the EIA document of 2004 (Gestion Aline Leclerc Inc., 2004a), accessibility being limited, the land is used only by industrial forest operators, aboriginal trappers and non-aboriginal users for sport hunting and fishing purposes. The mine site has only been accessible since the mining road was built in 1998. The operation of the mine would not affect the potential for land use.

The proponent's intended meetings with the relevant Aboriginal communities will provide information on their current and projected use of lands and resources for traditional purposes. The gathering of information to be carried out with the community organizations in the framework of the impact study of the Project will also document the current and projected use of the territory where the mine is located.

20.4.2.4 Archeology and Heritage

The area of the EIJB territory has a few recognized historic sites that are not conducive to large-scale movements, due in particular to the large wetlands (Gestion Aline Leclerc, 2004c). An evaluation of the archaeological potential of the site carried out by the specialized firm Archéo 08 highlighted the absence of constraints to the implantation of the mine in this aspect (Gestion Aline Leclerc, 2004c). The firm noted some potential along the banks of the Samson River. However, no work is planned in this sector in connection with the Fenelon Mine Property.

20.5 Mine Closure Requirements

A first conceptual closure plan was submitted to the MERN in December 2000 for the bulk sampling of 12,000 tonnes of ore with an open pit. In the context of the second bulk sampling, the closure plan has been revised three times, in April 2002, September 2002 and March 2003. In April 2004, the closure plan was reviewed to include the underground extraction of 220,000 tonnes of ore in order to start commercial production. All of the closure plan’s revisions were accepted by the MERN and C$35,000 was deposited as a warranty (70% of C$50,000).

Due to economic circumstances, the Project did not move into the exploitation phase in 2004 and has been under review since then. In accordance with the second paragraph of Article 232.6 of Quebec's *Mining Act* (L.R.Q., c. M-13.1) Wallbridge shall submit a revised plan to the Minister since amendments to the plan are justified by changes in the mining activities. Moreover, an amendment to article 111 of the Re*gulation respecting Mineral Substances other than Petroleum, Natural Gas and Brine* (Chapter M-13.1, r. 2) was adopted on July 23, 2013 (Decree 838-2013). Wallbridge must now provide a financial guarantee whose amount corresponds to the total anticipated cost of completing all the work set forth in its closure and rehabilitation plan.

The goal of mine site rehabilitation is to return the site to an acceptable condition, ensuring that the environment as a whole will eventually be able to take back its course. The closure plan focuses on the rehabilitation of land and areas affected by mining activities (i.e., roads, pads, buildings, water ponds, waste rock piles, etc.). The reclamation program includes the following:

- Pumping will stop and the open pit will be naturally flooded; the water table around the site is at the edge of the pit. Before flooding, the ramp access will be closed with waste rock, and a cement cap will be installed to cover the fresh air raise opening. The open pit will form a small lake with riprap slopes and will be surrounded by the peripheral road. Signs will be installed around the open pit to indicate the danger as required by the document *Guidelines for Preparing a Mining Site Rehabilitation Plan and General Mining Site Rehabilitation Requirements* (the "Guide");
- All buildings and infrastructure no longer required for post-rehabilitation monitoring will be dismantled. The material and equipment will be transported to recycling facilities. Waste material resulting from the dismantling operations will be transported to authorized sites for elimination;
- The overburden pile and waste rock piles will be reshaped for drainage before being vegetated;
- All the affected surface area of industrial site, including the waste rock piles, will be covered with the soil set aside during construction and then seeded;
- Contaminated soil will be treated onsite or disposed offsite in respect of the regulations;
- Water in the polishing pond will be analyzed and the sludge at the bottom of the pond will be characterized. Assuming the water quality meets the requirements of Directive 019, a ditch will be dug on the west side of the dike to connect the pond to the final effluent. The settling tower and outlet pipes will be removed. Assuming the sediments accumulated at the bottom of the pond meet the appropriate regulatory criteria, they will stay there. On the other hand, if they do not meet the standards, they will be transported to the bottom of the open pit. The dikes will be leveled towards the interior of the pond and the pond outlet will be towards the west diversion ditch.

The closure costs estimated by Gestion Aline Leclerc Inc. in 2004 were adjusted to include the updated aspects of the project and to comply with the current regulations and guidelines. It was also considered that all buildings, infrastructure and equipment were dismantled or rehabilitated, even if there is an opportunity to keep some components for further exploration needs. Thus, the estimated mine closure and rehabilitation cost of C$989,869 for the project on the Fenelon Mine Property presents a conservative scenario. This cost includes the direct (C$655,549) and indirect costs for site restoration, which include engineering costs (C$72,835) and post-rehabilitation monitoring (C$132,372), as well as a contingency of 15% (C$129,113). This estimate considers the waste rock to be non-acid generating and/or metal leaching. If the geochemical characterization concludes otherwise, the rehabilitation costs will have to be reviewed.

The cost increase compared to the 2004 estimate reflects the following factors: the more stringent regulation currently in place regarding post-closure requirements (i.e., the need to install danger signs around the open pit, the need to included fees for an environmental site characterization, and the obligation to include a minimum of 30% of the direct cost (at the conceptual stage) for engineering fees); the larger scope of the project (i.e., more infrastructure than in 2004); and the underestimation or omission of certain items (i.e., the dismantlement of the buildings and sedimentation pond, the levelling, the spreading and seeding of the overburden, the waste rock area, the surface infrastructure area and roads, and as the post-rehabilitation environmental monitoring). To comply with the MERN's guide on mine closure and rehabilitation (MERN, 2016), the mine closure and rehabilitation costs were calculated assuming no salvage value for the equipment and that a third party will complete the closure and rehabilitation work.

21 CAPITAL AND OPERATING COSTS

21.1 Basis of Capital and Operating Cost Calculations

The project on the Fenelon Mine Property constitutes a relatively small mine with a current short life span. For this reason, it is not the intent for Wallbridge to build an owner operations team or to purchase mining equipment. For the purpose of the pre-feasibility study, it is supposed that most of the onsite work would be completed by contractors. No processing activities, other than crushing, are planned on site; it is expected that the ore should be trucked to a mill for processing. A very small owner's team is envisioned to manage the onsite contractors and oversee the geology aspect of the project. Thus, capital and operating costs have been largely derived using contractor quotes from local contractors, and these have been validated based on our experience with similar projects. It is currently planned that most of the work should be split between the following main contracts:

- Mining contractor: install and operate all infrastructure required to dewater, develop and mine the orebody;
- Crushing / transportation contractor: manage the ore on surface, crush it and transport it to the offsite mill;
- Camp management contractor: provide the camp and manage it during the project; and
- Custom milling contractor: responsible for custom-milling of the Fenelon ore.

Some smaller contracts should be awarded for road maintenance and pond construction, for example.

21.2 Basis of Estimate

This basis of estimate section describes the methodology used for developing the capital and operating estimates for the project on the Fenelon Mine Property.

21.2.1 Accuracy

At the start of the study, it was intended that the pre-feasibility estimate was to be based on preliminary engineering data such as technical design criteria, basic layouts and service requirements, and preliminary equipment lists. Budget quotes were to be obtained for major equipment, while minor equipment costs were to be derived from similar, past or current projects performed by SNC-Lavalin, or by factoring the capacity and size of equipment. Layouts, quantities and preliminary take-offs were developed by SNC-Lavalin and provided to the contractors for the development of budgetary quotes Some factoring was required, and indirect costs were factored as a percentage of direct costs, obtained from allowances or based on typical industry "norms" or past project experience. The planned accuracy for the study was:

- Accuracy range: - 20% to + 30%;
- Nominal accuracy: ± 25%.

As the discussions with contractors progressed, more detailed quotes were provided and qualified by contractors as firm prices, which had an impact on the accuracy and contingency. A 3-D model was also produced to extract the underground mining quantities, contributing to improve the estimate accuracy.

21.2.2 Scope

This basis of the estimate covers the development of the following infrastructure:

- Underground mine, which should cover the following:
  - Mine development;
  - Mine dewatering;
  - Drift reconditioning;
  - Mine ventilation;
  - Underground facilities;
  - Definition drilling.

- Mine mobile equipment fleet;
- Mine infrastructure, which should cover the following:
  - Roads and pads;
  - Open pit and underground mine dewatering and sedimentation pond construction;
  - Mining camp;
  - Process water;
  - Ore and waste handling – crusher;
  - Compressed air;
  - Fuel;
  - Power;
  - Information technology;
  - Backfill;
  - Transportation to mill.

This estimate also includes for the construction and project indirect costs, royalties, and expected operational expenditures such as ore milling and refining.

21.2.3 Units of Measurement

The following metric units of measure are used for the estimate:

- m (metre) for linear distances (pipe runs, lateral development etc.);
- m^3 (cubic metre) for volumes (U/G excavations);
- m^2 (square metre) for areas (wire mesh, clearing etc.); and
- metric tons (tonnes) for ore/waste haulage, fabricated steel, etc.

21.2.4 Assembly of Overall Estimate

The estimate was assembled in Microsoft Excel. The estimate is broken down as per the work breakdown structure (WBS) provided in Appendix VI, and grouped under the following main categories:

- Pre-production;
- Capital costs;
- Operating costs;
- Remote camp operations;
- General and Administrative;
- Contingency; and
- Royalties.

21.2.5 Quantity and Cost Development

Table 21.1 clarifies how the quantities and costs were developed for each main category of the Fenelon Mine Property pre-feasibility study.

Table 21.1 – Estimate and cost basis

Main Category		
Pre-Production (Direct)	**Quantity Basis**	**Cost Basis**
Permits/Approval	Information on permit type/quantity provided by the owner.	Provided by the owner based on permits required.
Engineering	Estimated based on detailed engineering activities required for infrastructure and contractor support. Excluding detailed mining engineering to be performed by the owner as part of the owner's cost.	Cost based on similar projects and past experience.
Closure Costs	Information provided by a specialized consultant.	Cost estimated by a specialized consultant.
Capital (Direct)	**Quantity Basis**	**Cost Basis**
Polishing Pond	Semi-detailed engineering drawings and sections were produced by engineering.	Cost estimated based on similar projects and past experience.
Mine Access Road Upgrade	Allowance of material required based on assumed width and capacity of road to be upgraded.	Allowance per kilometre of road.
Underground waste development (ramp and level access)	Quantities for rock excavation for lateral development have been extracted from the Datamine Studio 5D Planner software (advanced model).	Composite costs were obtained from a general mining contractor chosen out of 3 bidders (firm prices).
Rehabilitation of Existing Workways	Quantities estimated from existing mine layout for level access.	General mining contractor estimated that 20% of the accesses will have to be rehabilitated.
Ventilation Raises and Escape Way / Site Setup	Quantities have been extracted from the Datamine Studio 5D Planner software (preliminary model).	Composite costs were obtained from a general mining contractor chosen out of 3 bidders (firm prices).
Mining Contractor Mobilization	Estimation is based on quantities extracted from the Datamine Studio 5D Planner software (preliminary model).	Composite costs were obtained from a general mining contractor chosen out of 3 bidders (firm prices).

Main Category		
Initial Dewatering – Pit and Underground	Detailed engineering drawings and sections were produced by engineering.	Composite costs obtained from a general mining contractor chosen out of 3 bidders (firm prices).
Operating (Direct)	**Quantity Basis**	**Cost Basis**
Underground Ore Development (including sills) / Contractor Indirect Costs	Quantities for rock excavation have been extracted from the Datamine Studio 5D Planner software (advanced model).	Composite costs were obtained from a general mining contractor chosen out of 3 bidders (firm prices).
Backfill stopes	Quantities have been extracted from the Datamine Studio 5D Planner software (preliminary model).	Composite costs were obtained from a general mining contractor chosen out of 3 bidders (firm prices) for most of the backfill (rockfill). Cemented backfill has been estimated based on historical data on similar projects.
Site Teardown / Site Demobilization	Estimation is based on quantities extracted from the Datamine Studio 5D Planner software (preliminary model).	Composite costs were obtained from a general mining contractor chosen out of 3 bidders (firm prices).
Crushing / Transportation to Mill / Milling / Delineation Drilling	Quantities were extracted from “Issued for Construction” drawings.	Composite costs were obtained from a general mining contractor chosen out of 3 bidders (firm prices).
Dewatering – Underground during Operations / Ventilation	Detailed engineering drawings and sections were produced by engineering.	Composite costs were obtained from a general mining contractor chosen out of 3 bidders (firm prices).
Refining	N/A	0.11% applied on total revenue (input from the owner).
Remote Camp Operation (Indirect)	**Quantity Basis**	**Cost Basis**
Selbaie Road Maintenance / Road and Site Maintenance	Quantities have been estimated from a survey that monitored the frequency and the type of vehicle circulating on the road / property.	Costs were obtained from a current awarded contract. Percentage is applied to calculate the shared road portion to be maintained by the owner.
Camp Setup-Demob / Camp Monthly Fee / Camp Catering and Janitorial	Quantities have been extracted from a preliminary Manpower Forecasting and Levelling (maximum 40 people) based on the site needs for the construction and operations phases.	Costs were obtained from a budgetary quote.

Main Category		
General & Administrative (Indirect)	**Quantity Basis**	**Cost Basis**
Owner's Cost	Quantities have been extracted from a preliminary Manpower Forecasting and Levelling based on the site needs for the construction and operations phases (by job position).	Costs obtained from the owner's internal job position salaries and fringe benefits.
First Nation Recurring Costs	N/A	Monthly costs given by the Owner.
Other Costs	**Estimate Basis**	**Cost Basis**
Duties & Taxes	N/A	These costs were calculated by a third party subject matter specialist firm.
Contingency	N/A	A deterministic contingency estimate exercise was performed. Each major element of the project was evaluated based on the level of details of the engineering and the cost estimating technique used.

21.2.5.1 Risk Analysis

There are no risk allowances in the estimate.

21.2.6 Exclusions

The following items are not included in the capital and operating cost estimate:

- Complementary studies;
- Project finance and interest costs;
- Sales taxes;
- Income taxes and duties (included in the financial model);
- Cost of sales and marketing;
- Legacy and site acquisition costs;
- Risk allowance;
- Dividends;
- Allowance for labour dispute or loss of time arising from strike or any disruption actions;
- Fluctuation to nominated currency exchange rates;
- Land acquisition costs;
- Project sunk (past) costs;
- Public relation program costs;
- Insurance costs;
- Allowance for schedule deceleration or acceleration;
- Corporate overheads; and
- Escalation.

21.3 Capital and Operating Expenditure

The estimated pre-tax capital and operating expenditures are summarized in Table 21.2.

Table 21.2 – Fenelon mining project cost expenditure summary (C$ ’000)

Cost Item	Q1 2017	Q2 2017	Q3 2017	Q4 2017	Q1 2018	Q2 2018	Q3 2018	Q4 2018	Total
Pre-production	300	668	80	0	0	0	0	448	1,496
Capital costs	0	103	1,588	1,766	1,728	53	0	0	5,238
Operating costs	0	105	1,120	3,124	7,838	5,845	4,982	697	23,710
Remote camp operations	0	0	678	400	537	537	450	438	3,041
General and administrative	0	0	299	433	567	702	567	299	2,866
Contingency	29	66	376	560	1,049	705	593	239	3,616
Royalties	0	0	0	8	248	289	234	31	809
Total	329	941	4,140	6,292	11,968	8,131	6,825	2,152	40,777

The total pre-tax project cost is C$40,777,243.

21.4 Contingency

Contingency is an integral part of the estimate and can best be described as an allowance for undefined items or cost elements that will be incurred, within the defined project scope, but that cannot be explicitly foreseen due to a lack of detailed or accurate information.

It should not be considered as a compensation for estimating inaccuracy nor is it intended to cover any costs due to potential scope changes, “Acts of God”, labour strikes, labour disruptions outside the control of the project manager, fluctuations in currency or cost escalation beyond the predicted rates.

Contingency is exclusive of project risk and exclusive of risk mitigation.

The estimated global contingency represents 9.9% of the total cost before contingency at P50. This percentage was determined by evaluating the quantity and cost precision of each system element of the cost estimate. As a result, contingency by item varies between 5.6% and 50%. In other words, each system element was assigned a precision level on quantity development and a precision level on cost origin.

The cost estimate accuracy falls within – 3% to + 18%.

21.5 Schedule

The following section defines the basis for the development of the project schedule and the life of mine plan.

21.5.1 Milestone

A milestone is a significant event in the schedule, such as an event restraining future work or an event marking the start or completion of a significant deliverable. A schedule milestone has no duration.

The study milestones were placed at the beginning of the schedule. This is to facilitate the reader's understanding of the schedule without having to scan through the entire document.

There are a total of six (6) milestones for the Fenelon mining project study and LOMP. Table 21.3 shows the project milestone dates. For the purpose of producing a financial evaluation, project dates were estimated based on the information available at the time of the report.

Table 21.3 – Main milestones of the Fenelon mining project

Milestone Id	Month
Engineering Start	-6
Permits are received	0
Mobilization Complete	1
Dewatering Complete	4
Mine Development Complete	11
End of Mining	17
Project Finish	18

21.5.2 Resources Planning

Resources have been loaded into the MS Project schedule to distribute key metrics such as underground development and ore production for each month of the project. The metrics have been used to calculate quarterly costs and inform the project cash flow balance sheet. Table 21.4 presents the main project metrics by quarter.

Table 21.4 – Mining metrics

Mining metrics						Total
Type	**Q4 2017**	**Q1 2018**	**Q2 2018**	**Q3 2018**	**Q4 2018**	
Ore production (DMT)						
Ore	1,789	30,997	29,513	29,208	5,216	96,721
Total	**1,789**	**30,997**	**29,513**	**29,208**	**5,216**	**96,721**
Horizontal development (m)						
Drift - Level Access	130	227	-	-		357
Drift and Fill Dev	-	33	58	35		127
Sill - Rock	184	294	246	-		724
Sill - Ore	51	220	157	-		428
Ramp	232	260	-	-		491
Total	**597**	**1,035**	**460**	**35**		**2,127**
Vertical development (m)						
Raise Development	-	50	-	-		50
Total	**-**	**50**	**-**	**-**		**50**

21.5.3 Duration

Duration is the time, in calendar days, required to complete an activity.

21.5.4 Calendar

There is only one calendar used in the schedule. It is based on 24 hours per day, 7 days per week, with no statutory holidays. No other project calendars were used.

21.5.5 Project Period

The project duration is defined by the date when the mine will reach 60% of its nominal production rate of 400 tpd. According to the schedule, the mine starts to generate ore at a rate of 240 tpd in Q1 2018.

21.5.6 Assumptions

The start date of the project is assumed to be February 11, 2017, and the permits are assumed to be received by July 1, 2017.

The Project schedule has been developed on the basis of the following:

- Daily ore production of 400 tonnes per day;
- Mine development rate of 6 metres per day (multiple headings);
- Stope drilling quantities of 3.0 tonnes per metre drilled;
- Stope drilling rates of 200 metres per day; and
- Backfill duration is assumed to be the same as mucking duration.

21.5.7 Path of Execution

The reception of the permits is the major milestone that should constitute the start of the project execution. According to the information obtained at the moment this report was written, the assumed date for the receipt of the permits is July 1, 2017.

Regardless, there are a number of activities that can be completed prior to the permits being received. The detailed engineering and the procurement activities can all be completed prior to July 1, 2017. These activities were scheduled as late as possible in order to define the start of the project. The start of the project is expected to be February 11 such that the detailed engineering and procurement are completed, permits are received, and the contractors can mobilize July 2, 2017.

The polishing pond upgrade needs to start right away and be completed for the dewatering of the pit and the underground mine to start.

It is anticipated that the construction of the surface infrastructure, including the camp, should be completed prior to the underground mine being dewatered. The rehabilitation of the underground mine openings can be done while the mine is being drained.

It is anticipated that the mine should have two development crews, one for the development of the ramp and level access (crew #1) and the other for the development of the sills (crew #2).

The focus of Crew #1 will be ramping down to the bottom of the orebody on the 5150 Level as quickly as possible, while crew #2 develops the upper levels to start mining and generate revenue as soon as possible (early muck).

Once the ramp has reached the 5150 Level, crew #2 will be relocated on this level in order to start mining the orebody bottom-up. Mining will continue until all the ore is depleted. The mine will be decommissioned, the operation will cease and the site returned to nature.

21.5.8 Schedule Contingency

There is no contingency built into the schedule at this moment.

21.5.9 Risk and Opportunities

The following section shows the risk and opportunities that pertain to the Fenelon Mine Property. The risks have been identified but not quantified in terms of costs or schedule.

21.5.9.1 Risk

- Conditions of the existing underground mine are unknown; therefore, the rehabilitation process may be both a financial and scheduling risk;
- Slope conditions of the open pit while it is being dewatered remain unknown. It is assumed that the work recommended by Golder in 2004 regarding the stability of the open pit slopes was performed by the former site operator and that no slope stabilization work will be required once the mine is dewatered and the slopes exposed;
- Dewatering will take place while the mine is in operation, therefore water infiltration should be further monitored during the early stages of operation and the dewatering system should be modified if required; and

- Currently, several users share the costs of the Selbaie road maintenance. There is a risk that some users would no longer use this road, which would make the costs of maintenance for the road to Fenelon mine could go up.

21.5.9.2 Opportunities

- Assuming the permit to take water (PTTW) is received early, the mine could be dewatered sooner, providing a better understanding of site conditions;
- Additional ore could be discovered underground;
- Dewatering of the mine can also be done through the ventilation raise; and
- The issuer is an exploration company and has loss carry forwards which can be applied to reduce income taxes.

21.5.10 Scheduling Techniques and Software

MS Project was used as the scheduling tool. There is only one task with a constraint and that is the “Permits are received” task with a “must finish on” date. All other activities have predecessor and successor activities tied to them.

There are five user fields that were created inside MS Project to help filter the schedule as per the work breakdown structure and by crew to help with resource loading and data exportation from the schedule on to the project financial evaluation sheet. The project schedule is presented in Appendix VII.

22 ECONOMIC ANALYSIS

22.1 Summary

Based on the current assumptions, discounted cash flow modelling of the project yields a pre-tax NPV of C$5.84 million at a 5% discount and a pre-tax internal rate of return ("IRR") estimate of 92%. The NPV and IRR after income taxes and before any withholding tax are C$2.80 million and 60%, respectively. A summary of these results is presented in Table 22.1.

Table 22.1 – Base case estimated financial results

Pre-tax	
NPV at 5% Discount Rate (C$ '000)	5,842
Internal Rate of Return (IRR)	92%
Payback Period	Q3 2018
After-tax	
LOM NPV at 5% Discount Rate (C$ '000)	2,802
Internal Rate of Return (IRR)	60%
Payback Period	Q2 2018

22.2 Commodity Prices

For the reserves and all engineering studies the determination of the price of gold was established by InnovExplo and it was decided to use the September 2016 6-month trailing average of US$1,285. This is included in Table 15.1. This 6-month trailing average exchange rate of 1.31 is also used in the cut-off grade determination.

22.3 Financial Analysis

An after-tax model was developed for the Fenelon Mine Property pre-feasibility study (see Appendix VIII). All costs are in Q4 2016 Canadian dollars with no allowance for inflation or escalation. The financial analysis first period is Q1 2017.

Wallbridge owns a 100% interest in the Fenelon Mine Property, with a 1% NSR royalty payable to Balmoral and Cyprus (each) in the event of commercial production. In the cash flow analysis, this royalty was considered on all ounces produced from the Property.

The economic evaluation was performed by the Internal Rate of Return (IRR) and the Net Present Value (NPV) methods using estimates of capital and operating costs, a construction schedule, a production schedule and estimates of future gold ore prices provided as mentioned in previous sections of this report. Since the financial analysis is based on a cash flow and schedule estimates, it should be expected that actual financial results will differ from these predictions.

The IRR on an investment is defined as the rate of interest earned on the unrecovered balance of an investment. The discount rate makes the NPV of all cash flows equal to zero.

The NPV method converts all cash flows for investments and revenues occurring throughout the planning horizon of a project to an equivalent single sum at present time at a specific discount rate. The discount rate used as a base case in the analysis is 5%. According to the NPV method, a positive NPV represents a profitable investment where the initial investment plus any financing interest are recovered.

22.4 Taxation

The project on the Fenelon Mine Property is subject to federal and provincial income taxes and taxes relating to Québec mining rights. All taxes were calculated by a Québec-based mining tax expert.

Income taxes are calculated in accordance with the federal and provincial tax legislations relating to mining companies. The combined federal income tax and mining taxes applicable to the duration of the project was estimated to be C$3,365,655.

The estimated taxes may be reduced if deductibles, such as exploration assets and expenditures, can be used to reduce the project's income during the operation of the mine. Approximately C$6,000,000 in losses carried forward would bring down the total income tax amount to C$1,763,655 (reduction of C$1,602,000).

22.5 Assumptions and Results

The main parameters and cash flow analysis results for the entire project are presented in Table 22.2.

Table 22.2 – Base case – Assumptions and results

Parameters	Results
Current estimated Mineral Reserves (Proven and Probable)	96 720 @ 9.3 g/t Au
Mill recovery	97%
Life of mine (LOM)	14 months
Daily mine production	400 tpd nominal
Total ounces mined over LOM	28,922 oz
Gold recovered over LOM	28,054 oz
Gold price (US$)	$1,285.28
Exchange rate (CAD/USD)	1.31
Gold price (C$)	$1,689.41
Total gross revenue (C$)	$47,395,584
Total expenditure (pre-tax, excluding royalties, C$)	$39,967,968
Royalties	1% NSR payable to Cyprus and Balmoral each on all ounces produced from the Fenelon Mine Property.
Total expenditure (pre-tax, including royalties, C$)	$40,777,243
Average all in cost per tonne (C$)	$421.60 per tonne
Average all in cost per ounce (US$)	$1,105.79 per ounce
Pre-tax	
NPV at 5% discount rate (C$ '000)	5,842
Internal rate of return (IRR)	92%
Payback period	Q3 2018
After-tax	
LOM NPV at 5% discount rate (C$ '000)	2,802
Internal Rate of Return (IRR)	60%
Payback Period	Q2 2018

22.6 Overall Cash flows

Details of the cash flow analysis are presented in Figure 22.1 on a quarterly and cumulative basis (in constant dollars, pre-tax).

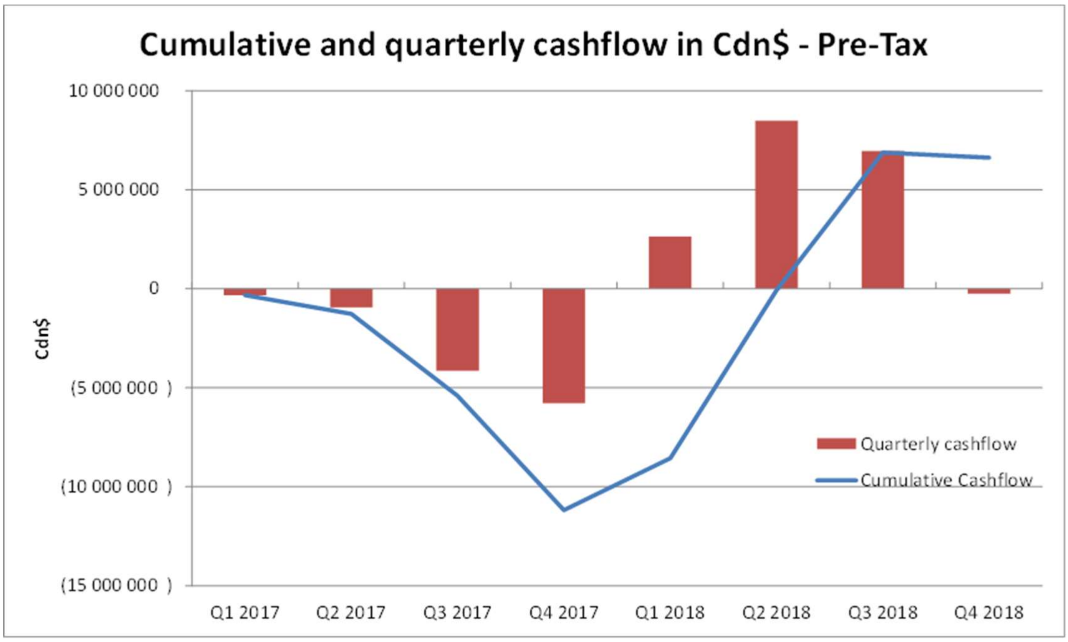


Figure 22.1 – Quarterly and cumulative cash flow

22.7 Sensitivity Analysis

The sensitivity of the Pre-tax Project NPV to changes in key variables is shown in Tables 22.3 to 22.5. The pre-tax Project NPV appears to be most sensitive to changes in the estimates of the gold prices and least sensitive to changes in costs. The selected input parameters used in the sensitivity analysis are:

- Selected gold prices;
- Selected discount rates;
- Projects costs;
- Canadian to US exchange rate.

Table 22.3 – Pre-tax NPV for varying gold prices and discount rates (C$ '000)

Gold price (C$/oz)	1,689 (Base Case)	1,400	1,500	1,600	1,700	1,800
Discount Rate						
Pre-tax NPV at 0%	6,618	-1,330	1,416	4,163	6,909	9,655
Pre-tax NPV at 5% (Base Case)	5,842	-1,633	950	3,532	6,115	8,698
Pre-tax NPV at 10%	5,155	-1,895	541	2,977	5,413	7,849

Table 22.4 – Pre-tax NPV for varying project and operation total expenditures (C$ '000)

Total expenditures excluding royalties	Value	Pre-Tax NPV @ 5% discount rate
-10%	$35,971	$9,638
Base Case	$39,967	$5,841
10%	$43,964	$2,045

Table 22.5 – Pre-tax NPV for different exchange rates (C$ '000)

Exchange rate CAD to USD	Pre-Tax NPV @ 5% discount rate (C$ '000)
1.20	$2,043
1.31 (Base Case)	$5,841
1.40	$8,681

23 ADJACENT PROPERTIES

23.1 Detour East Property (Balmoral Resources Ltd)

The following description of the Detour East Property was taken and modified from the September 30, 2015 Management's Discussion and Analysis (MD&A) report filed by Balmoral Resources Ltd on SEDAR.

The Detour East Property (Fig. 23.1) covers more than 20 kilometres of the Sunday Lake, Detour Lake and Lower Detour Lake deformation zones, stretching east from the Québec-Ontario border. The property consists of 539 mining claims (approximately 21,172.71 ha) held 100% by Balmoral, and an additional 18 mining claims (approximately 997.54 ha) in which Balmoral holds a 69% joint venture interest (the remaining 31% being held by Encana Ltd). Balmoral is the project operator. The Detour East Property is located immediately east of the Detour Lake mine.

Geochemical surveying was completed on the property during the fourth quarter of 2014, highlighting several areas and trends for further follow-up. Balmoral also located drill core from a number of historical drill holes completed on the Detour East Property; the company has taken control of the core and transported it to the Fenelon Camp. Detailed re-logging of these holes was pending at the time of the MD&A report date. Balmoral completed a single drill hole on the southwestern part of the Detour East Property in the summer of 2015 that intersected two intervals of weakly anomalous gold mineralization in a large gabbro complex.

23.2 Casault Property (Midland Exploration Inc.)

The following description of the Casault Property was taken and modified from the 2015 Annual Report filed by Midland Exploration on SEDAR. Midland Exploration holds a 100% interest in the Casault Property (Fig. 23.1). At the end of 2014, this property consisted of 300 claims covering an area of approximately 16,507 ha.

In winter 2015, a drilling program consisting of seventeen (17) holes for a total of 3,467.2 metres was completed in partnership with SOQUEM (50/50 JV). This program targeted the most promising gold occurrences discovered in 2012–2013. These areas include the north contact of the Turgeon Pluton, where drill hole CAS-12-07 returned 10.4 g/t Au over 1.45 m, as well as areas immediately north and west of the conglomerate basin where pyrite and jasper clasts were identified in 2013. In the northern area, drill hole CAS-13-28A ended in a gold-bearing zone grading 0.29 g/t Au over 9.0 m. Two holes were also completed to test IP anomalies on the central block.

An IP-Orevision survey was also completed in the winter of 2015 (South Grid). This 17.1-km survey identified several strong chargeability responses near the granodiorite contact. These anomalies correspond to the mineralized package (sediments and diorite intrusions) found between the Turgeon Pluton and the mafic volcanics. Two drill holes (CAS-15-47 and 48) were completed to test this IP axis.

Another IP-Orevision survey was completed in March 2015 on the North Grid. This grid totalled approximately 25 kilometres. Several new IP anomalies were identified on the North grid.

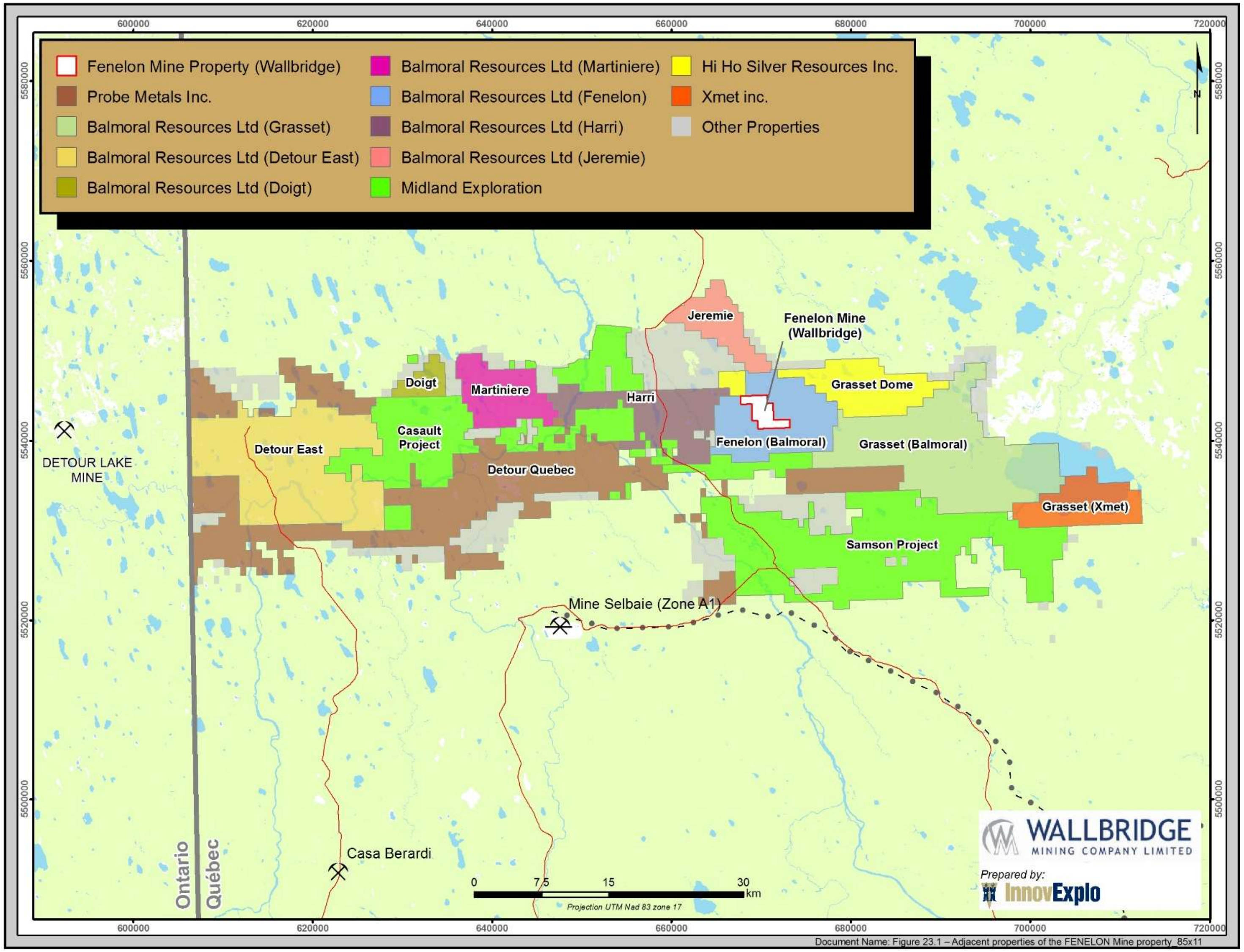


Figure 23.1 – Wallbridge's Fenelon Mine Property and adjacent properties along the Sunday Lake Deformation Zone in the province of Québec

During the 2015 summer drill program, fifteen (15) drill holes totalling 5,002.00 metres were completed in partnership with SOQUEM (50/50 JV). Five (5) of these, CAS-15-55 to CAS-15-59, were drilled in the area of the gold-bearing porphyry intrusion that had been followed up in drill hole CAS-15-44 the previous winter. These five holes, spread over a distance of 2 kilometres, intersected several anomalous gold values associated with porphyry intrusions and gabbro locally altered to silica, sericite and hematite, thereby confirming the excellent gold potential of the area, which is strategically located in a folded zone at the contact between Timiskaming-type basin conglomerates and mafic volcanics. In addition, new anomalous zones were intersected for the first time in the mafic volcanics along the northern contact of the porphyry intrusion. Anomalous gold-bearing zones running less than 0.50 g/t Au over 0.5 m or more were intersected in this area.

The other drill holes completed during this program to test geological, structural, IP and TDEM targets did not return significant gold values despite the fact that all targets were explained by the presence of sulphides.

23.3 Doigt Property (Balmoral Resources Ltd)

The following description of the Doigt Property was taken and modified from Balmoral's website.

Balmoral holds a 100% interest in the Doigt Property (Fig. 23.1). Balmoral acquired the Doigt Property by staking in late 2011. The Doigt Property covers a roughly 5 by 5 kilometre block of volcanic- and intrusive-dominated stratigraphy to the west of the northern end of the Martiniere Property, and about 6 kilometres northwest of Balmoral's Bug Lake and Martiniere West gold discoveries.

Work to date has been primarily focused on understanding the geology and mineral potential of the Doigt Property. The Doigt Property is located in the Casault structural domain, which is sandwiched between the Detour and Martiniere structural domains to the west and east, respectively.

The Doigt Property is the least explored portion of the Detour Trend Project, with only two drill holes known on the property, both completed by Balmoral in 2013. Balmoral's first two drill holes intersected narrow intervals of anomalous, structurally controlled gold mineralization, thereby confirming the potential for mesothermal gold mineralization on the Doigt Property. Given the property's distance to regionally significant deformation corridors, targeting should focus on secondary structural corridors, in particular where these intersect known lithological contacts.

To date no indication of significant base metal potential has been observed on the Doigt Property. A narrow zinc-copper bearing vein was intersected in one of the two holes drilled on the property but does not appear to have any significant lateral extent. Additional surface mapping may aid in further understanding the property and determining the potential for base metal mineralization.

23.4 Martiniere Property

The following description of the Martiniere Property was taken and modified from the September 30, 2015 MD&A report and the 2014 Annual Information Form report filed by the issuer on SEDAR, as well as from information on Balmoral's website.

Balmoral owns a 100% interest in the Martiniere Property (Fig. 23.1), which hosts a number of near-surface occurrences of gold mineralization, including the West, Central and Bug Lake zones (or trends). The Bug Lake Trend is a structurally-controlled orogenic gold prospect hosted by the Bug Lake Fault Zone (BLFZ), which was recognized as a significant structure as early as 2011 but not identified as a gold-bearing trend until the summer of 2012. Similar to deposits throughout the Abitibi region, this discovery is characterized by high gold grades, variable widths and strong silica-carbonate alteration. The Bug Lake Trend remains open for expansion, but has been traced thus far across 1,200 metres of strike length and to vertical depths of over 400 metres.

Located 600 metres west of the central portion of the Bug Lake Trend, the West Zone is a second prominent high-grade gold-bearing feature. Originally discovered by Cyprus Canada in the late 1990s, Balmoral has drill-defined the West Zone for 400 metres along strike and to vertical depths of over 300 metres. The West Zone sits in a separate structural zone from Bug Lake. This shear zone also hosts a number of gold occurrences on the Martiniere Property that warrant additional examination.

In addition to these two gold zones, Balmoral has identified at least 10 other prominent gold occurrences on the Martiniere Property, the most recent of which is some 2.0 kilometres east of any previous gold-bearing intercepts. In addition, the historical Norbug gold occurrences, located more than 3 kilometres to the northeast of the heart of the Bug Lake Trend, suggest the presence of a large gold-bearing system in the greater Martiniere area, only a small portion of which has been tested to date.

Balmoral is principally focused on delineating a number of zones of gold mineralization along the Bug Lake Trend that were discovered in 2012. Gold mineralization along the Bug Lake Trend (the Upper and Lower Bug Lake, Bug Lake Footwall and Bug Lake Hanging Wall zones) is localized along an early-stage fault system that was reactivated multiple times and which locally features high gold grades. Drilling to date on the Bug Lake Trend has intersected significant gold mineralization for over 1,800 metres along strike and to vertical depths of 400 metres.

The summer and winter 2015 drill programs focused on infill drilling in the northern half of the Bug Lake Trend at shallow depths between surface and 250 metres vertical depth. Results were highlighted by a number of high-grade intercepts, including 19.55 g/t Au over 44.45 m from the Bug Lake Footwall Zone (see Balmoral's news release of April 20, 2015). On May 13, 2015, Balmoral released additional results from the winter program, including a follow-up intercept of 9.30 m grading 15.75 g/t Au from the Bug Lake Footwall Zone and a series of broad gold mineralized intercepts from the Upper and Lower Bug Lake Zones. Summer drill results included the intersection of Bug Lake-style gold mineralization 600 metres beyond the previous southern limit of Bug Lake Trend.

Drilling has also begun to delineate a new gold-bearing structural zone on the Martiniere Property. Two holes, one drilled in late 2014 and a second completed this summer approximately 185 metres further east, have intersected three subparallel zones of gold mineralization in a corridor more than 200 metres wide, characterized by moderate deformation and dyking. These new discoveries are approximately 2.3 kilometres west of the northern end of the Bug Lake Trend.

Balmoral has retained a consultant to assist with metallurgical testing of a bulk sample from the Bug Lake Zone. There are no current resources calculated for the Martiniere Property.

In 2011, Balmoral also reported the discovery of a volcanogenic massive sulphide ("VMS") system on the Martiniere Property. Balmoral intersected a narrow, strongly brecciated interval near the upper margin of the Martiniere East VMS system (see Balmoral's news release of December 5, 2011). Hole MDE 11-09 intersected 0.50 m grading 0.72% Cu, 0.74% Zn, 1,390.0 g/t Ag, 74.60 g/t Au and 1,850 ppm W. The extremely high-grade gold-silver breccia intersected in hole MDE 11-09 sits in the immediate footwall to the massive sulphide portion of the Martiniere VMS system in this hole.

Drilling in the winter of 2015 (see Balmoral's news release of April 20, 2015) intersected semi-massive sulphides believed to be associated with this discovery, which yielded copper, zinc, gold and silver assay results of potential economic interest. Hole MDE 15-172 intersected 2.10 m grading 1.52% Cu, 4.20% Zn, 29.44 g/t Ag and 2.79 g/t Au from a semi-massive sulphide interval incorporated into a brecciated phase of the Upper Bug Lake Gold Zone.

23.5 Harri Property (Balmoral Resources Ltd)

The following description of the Harri Property was taken and modified from Balmoral's website.

Balmoral owns a 100% interest in the Harri Property (Fig. 23.1). The Harri Property covers a 20-kilometre stretch of volcanic and sedimentary stratigraphy located immediately north of and along the Detour Lake and Sunday Lake deformation zones, located between Balmoral's Martiniere and Fenelon properties. Balmoral acquired the Harri Property by staking in late 2010 and 2011. Work to date has primarily focused on understanding the geology and mineral potential of the Harri Property.

The Harri Property traces the northern margin of the Sunday Lake Deformation Zone for approximately 20 kilometres in an east-west direction across the property. The Harri Property also covers the eastward extension of the structural/stratigraphic sequence hosting the Martiniere gold system on Balmoral's adjacent property to the west. Across the Harri Property, the Sunday Lake Deformation Zone and its related structures are sparsely tested and have not been well understood historically due to the heavy overburden cover.

The southern portion of the Harri Property hosts a highly unusual, dome-shaped inlier of sedimentary stratigraphy approximately 10 kilometres across. This highly unusual formation is ringed by an extensive series of EM conductors. Historical drilling in this area has been directed mainly at VMS (Zn-Cu) targets with limited success. The stratigraphy in this area is poorly understood.

23.6 Grasset Property (Balmoral Resources Ltd)

The following description of the Grasset Property is taken from the summary contained in the Grasset Technical Report (Richard and Turcotte, 2016), dated January 12, 2016.

Balmoral owns a 100% interest in the Grasset Property (Fig. 23.1). The Grasset Property is not subject to any royalty, back-in right, or other agreement or encumbrance.

The Grasset Property hosts the Grasset deposit located in the Grasset Ultramafic Complex ("GUC"). The GUC formed by a stacked piles of basalts, gabbro and ultramafic sills and dykes, with minor rhyodacitic to dacitic volcaniclastics and rhyolite flows, and several narrow intercalated bands of iron formation and graphitic argillite in apparent conformable contact with the overlying rock units. The general attitude of the GUC is WNW, pinched between the Jérémie Pluton and the Opatica Subprovince. Several zones of ductile deformation have been intercepted in drill holes along strike in the complex, suggesting that the NW-SE trend may correspond to a major fault, parallel to other similar faults north and south of the Sunday Lake Deformation Zone. The southern portion of the complex is sheared and possibly folded by the deformation zone.

Mineralization of Grasset deposit is concentrated in two stacked sulphide-bearing horizons (H1 and H3) oriented NW-SE within vertically dipping peridotite ultramafic units. Mineralization consists of metre-scale layers of net-textured, blebby semi-massive and massive sulphides. Pyrrhotite is the dominant sulphide mineral, with subordinate amounts of pentlandite, chalcopyrite and pyrite. The concentration of pentlandite and chalcopyrite is proportional to the total sulphide content. The two horizons are stacked 25 to 50 metres thick and separated by 10 to 50 metres of unmineralized ultramafic rock. Horizon 3 (H3) is defined over a strike length of roughly 500 metres, and hosts the bulk of the high Ni-Cu-PGE values defined to date. Horizon 1 (H1) has been defined over a longest strike length (~900 m) and hosts moderate nickel grades (<1%) over its entire extent. Both zones are open at depth.

On March 7, 2016, Balmoral reported the initial resource estimate on the Grasset deposit. This initial independent resource estimate for the Grasset deposit was prepared by InnovExplo (Richard and Turcotte, 2016). At a 1.00% NiEq cut-off grade, the H3 + H1 zones contain a combined resource as follows:

- Indicated Resource: 3.45 Mt at 1.79% NiEq, corresponding to 1.56% Ni, 0.17% Cu, 0.03% Co, 0.34 g/t Pt and 0.84 g/t Pd; which equates to 136,279,000 lbs NiEq.
- Inferred Resource: 91,100 t at 1.19% NiEq, corresponding to 1.06% Ni, 0.11% Cu, 0.02% Co, 0.20 g/t Pt and 0.48 g/t Pd; which equates to 2,393,900 lbs NiEq.

The current mineral resource estimate is based on results from 111 DDH (39,999 m) completed by Balmoral since 2014. The base case current resource is reported above a 1.00% NiEq cut-off grade after incorporation of estimates for mining recoveries, mining dilution, milling recoveries, smelting and refining charges and certain penalties, as well as estimated operating costs based on those associated with mines currently operating in the local region.

The majority of the Resources are contained within the steeply plunging core of the H3 zone from surface to a vertical depth of approximately 550 metres. This core zone remains open to depth for potential expansion.

The recent drilling by Balmoral (2011 to 2014) also outlined gold mineralization, named the Grasset Gold discovery, at the contact between the sequence of strongly deformed polylithic Timiskaming-type conglomerates and a mafic intrusive of the Manthet Group, in the footwall of the Sunday Lake Deformation Zone. The first hole intersected 33.00 m grading 1.66 g/t Au, including two higher grade intervals grading 6.15 g/t Au over 4.04 m and 4.18 g/t Au over 5.00 m. The mineralization is hosted in an anastomosing quartz-carbonate vein system along the contact, which is open laterally and at depth.

23.7 Fenelon Property (Balmoral Resources Ltd)

The following description of the Fenelon Property was taken and modified from the September 30, 2015 MD&A and March 31, 2016 MD&A filed by Balmoral on SEDAR, as well as from information on Balmoral's website.

Balmoral owns a 100% interest in its Fenelon Property (Fig. 23.1). In January 2013, Balmoral completed the acquisition of a 100% interest in the property from Cyprus Canada and granted a 1% NSR on the property in favour of Cyprus Canada as required by the acquisition agreement.

During the first quarter of 2015, Balmoral commenced drill-testing of several geophysical anomalies along the projected northwestern continuation of the Grasset Ultramafic Complex through its property. The target was Ni-Cu-PGE mineralization similar to that recently discovered on its adjacent Grasset Property. Four new Ni-Cu-PGE occurrences were identified, highlighted by an intercept grading 0.37% Ni, 0.05% Cu, 0.06 g/t Pt and 0.13 g/t Pd in hole FAB 14-46, located 6.5 kilometres northwest of the Grasset discovery. In addition, high-grade gold mineralization grading 216 g/t Au over 0.76 m was discovered in hole FAB 15-50, along the northeastern contact of the Grasset Ultramafic Complex, near nickel sulphide mineralization.

During the first quarter of 2016, Balmoral completed two holes targeting geophysical anomalies on its Fenelon Property with no significant results reported.

On May 25, 2016, Balmoral entered into a Letter of Intent to sell to Wallbridge its interest in a 10.5-km^2 subdivision of Balmoral's Fenelon Property, which became the issuer's Fenelon Mine Property.

23.8 Jeremie Property (Balmoral Resources Ltd)

The following description of the Jeremie Property was taken and modified from the September 30, 2015 MD&A filed by Balmoral on SEDAR, as well as from information on Balmoral's website.

Following the discovery of Ni-Cu-PGE mineralization at Grasset, Balmoral acquired, by staking, a 100% undivided interest in a new property to the north of its Fenelon Property (Fig. 23.1).

The Jeremie Property covers a series of highly magnetic rocks, beneath extensive overburden cover. The rocks are interpreted as the northwestern extension of the Grasset Ultramafic Complex.

Limited historical drilling on the property has identified low-grade nickel mineralization, suggesting potential for VMS and gold discoveries. Work by a predecessor company in 2006–2007 identified a number of Cu-Zn-Ag-Au occurrences within this felsic volcanic sequence on its adjacent Fenelon Property.

In the winter of 2015, Balmoral completed a winter exploration trail into the Jeremie Property to facilitate initial drill testing of several geophysical targets along the projected extension of the Grasset Ultramafic Complex during the second quarter of 2015. Two targets were tested but failed to intersect ultramafic lithologies. Anomalous zinc mineralization was intersected over narrow intercepts in both holes. Two holes completed on the property in the summer of 2015 intersected mafic volcanic and intrusive rocks and minor iron formation. No significant mineralization was obtained in either hole.

While not considered as highly prospective for gold as it is for base metals, Balmoral does recognize some potential for mesothermal gold mineralization on the property associated with structural zones adjacent to both ultramafic rocks of the Grasset Ultramafic Complex and the larger Jeremie batholith.

23.9 Detour Québec Properties (Adventure Gold Inc.)

The following description of the Detour Québec Properties was taken and modified from the October 31, 2015 MD&A report filed by Adventure Gold Inc. on SEDAR, as well as from information on Adventure Gold's former website.

On June 10, 2016, Probe Metals completed the acquisition of Adventure Gold Inc.

The Detour Québec Project comprises nine (9) properties (Fig.23.1) totalling more than 816 claims and covering an area of 45,304 ha (453 km^2). The properties are strategically located over a strike length of 80 kilometres on the Detour Gold Trend, which encompasses the Detour Lake mine.

In recent years, Adventure Gold had explored its Detour Québec Project using IP surveys, ground magnetic surveys and helicopter-borne electromagnetic VTEM-MAG surveys. This exploration work highlighted promising areas where many geophysical anomalies (from IP and VTEM surveys) near strong gold anomalies were identified as potential new gold-bearing zones along the Sunday Lake, Massicotte and Lower Detour/Grasset deformation zones and other subsidiary fault zones (see the Adventure Gold website for details). A compilation of previous work also highlighted follow-up drilling targets along the proven gold structures close to positive historical drilling intercepts and grab samples. The best targets include near-surface follow-up drilling on historical intercepts grading 3.7 g/t Au over 4.0 m, 18.3 g/t Au over 1.1 m and 3.7 g/t Au over 3.1 m. Each area contains quality IP anomalies and/or follow-up drilling targets, and warrants new drilling. Historically, very little exploration work has been done on these claims, and only limited drilling has been carried out on one area with VMS-style gold, zinc and copper mineralization. This geological environment shows some similarities with the Martiniere Property located further east.

23.10 Samson Property (Midland Exploration Inc.)

The following description of the Samson Property was modified from the 2015 Annual Report filed by Midland Exploration on SEDAR, and from other information on the Midland Exploration website.

Midland Exploration holds a 100% interest in the Samson Property (Fig. 23.1). The Samson Property consists of 551 claims covering a surface area of about 30,592 hectares. In December 2014, the aim of a major ground-based geophysical program, totalling about 60 kilometres and including magnetic and ground EM surveys, was to characterize a series of untested MegaTEM conductors coinciding with strong magnetic responses. About a dozen high-priority MegaTEM targets were selected for this ground follow-up due to their association with strongly magnetic units interpreted as ultramafic rocks. Following the TDEM-ARMIT survey conducted over the best MegaTEM conductors, six (6) conductors were selected for drilling. In the summer of 2015, six (6) DDH totalling 1,625.5 metres were completed on the Samson Property to test the selected TDEM-ARMIT conductors. Only anomalous values in copper, nickel and gold were reported by Midland Exploration for this drilling program.

23.11 Grasset Property (Xmet Inc.)

The following description of the Grasset Property taken and modified from information on the Xmet Inc. website.

The Grasset Property (Fig.23.1) is 100% owned by Xmet Inc. through its wholly-owned subsidiary Duquesne-Ottoman Mines Inc. The property comprises 128 contiguous exploration claims totalling 7,040 hectares.

The property has seen relatively little exploration work. Fourteen (14) drill holes were collared on the claims between 1959 and 1987 for a total of 1,910 metres. All holes were drilled from land; no holes were collared on Lac Grasset. A few geophysical surveys were undertaken, consisting mainly of magnetic/gradiometric and EM surveys.

Two mineral occurrences have been identified on the property: Ingamar (0.93 g/t Au over 1.83 m) and Harricana-Turgeon (0.50% Cu over 1.0 m). Both occurrences occur along the south shore of the lake. On the western shore of the lake, a few hundred metres from the property boundary, a Cu-Au showing is reported to have assayed 5.5 g/t Au in grab sample (Longley, 1943). The Detour Lake–Sunday Lake Deformation Zone is also interpreted to cross the claims near the south shore of Lac Grasset.

23.12 Grasset Dome Property (Hi Ho Silver Resources Inc.)

The following description of the Grasset Dome Project was taken and modified from information on the Hi Ho Silver Resources Inc. website.

Hi Ho Silver Resources Inc. (“Hi Ho”) holds a 100% interest in the Grasset Dome Property, which covers approximately 6,000 hectares adjacent to Balmoral’s Grasset Property. The property is prospective for Ni-Cu-PGE deposits, gold deposits and copper-zinc-gold-silver VMS deposits.

Hi Ho is planning a geological and geophysical evaluation of the property based on available data in anticipation of an exploration program this season. The property is accessible by logging roads in well-drained terrain which has been largely logged-over in recent years.

On February 10, 2015, Hi Ho announced that it has purchased an additional eleven (11) mineral tenures covering 605 hectares that were added to Hi Ho's Grasset Dome Property.

23.13 Gold and Base Metal Potential of Adjacent Properties

InnovExplo has not verified the above information about mineralization on adjacent properties around the issuer's Fenelon Mine Property. The presence of significant mineralization on these properties is not necessarily indicative of similar mineralization on the Fenelon Mine Property.

24 OTHER RELEVANT DATA AND INFORMATION

All relevant data and information regarding the issuer's Fenelon Mine Property have been disclosed under the relevant sections of this report.

25 INTERPRETATION AND CONCLUSIONS

25.1 Interpretation

25.1.1 Background

The objective of the author's mandate was to prepare a Technical Report (the "Report") to present and support the results of a Pre-feasibility Study ("PFS") for the Fenelon Gold Mine in accordance with Canadian Securities Administrators' National Instrument 43-101 Respecting Standards of Disclosure for Mineral Projects ("NI 43-101") and Form 43-101F1.

The authors consider the present Pre-feasibility Study (and Resource Estimate herein) to be reliable and thorough, based on quality data, reasonable hypotheses, and parameters compliant with NI 43-101 and CIM standards regarding mineral resource estimates.

The issuer's Fenelon Mine Property covers 1,052 hectares and is located in west-central Québec about 75 kilometres northwest of the town of Matagami. Geologically, it is situated near the Sunday Lake Deformation Zone, which hosts the Detour Lake mine in Ontario (Detour Gold Corporation) and the Martiniere gold project in Québec (Balmoral Resources Ltd). The Fenelon deposit (a.k.a. the Discovery Zone) has seen both underground and open pit development in the past.

In all, more than 50,000 metres have been drilled on the Fenelon Mine Property, and two bulk samples have been mined and processed from the deposit. In 2001, a 13,713-tonne bulk sample mined from a small open pit was test-milled at the Camflo Mill in Malartic. The sample returned 132,039 grams (4,245 oz) of gold for a reconciled head grade of 9.84 g/t Au using a calculated recovery of 97%. A second bulk sample, consisting of 8,073 tonnes mined from underground, was also milled at Camflo and returned 80,731 grams (2,596 oz) of gold for a reconciled head grade of 10.7 g/t Au and a gold recovery of 93.5%. Compensating for the operational problem that occurred during the ore processing of this second bulk sample, the gold recovery would have been in the range of 97%.

Prior to the 2016 mineral resource estimate, resources had last been estimated in September 2004 and updated in January 2005. About 16,000 metres of additional diamond drilling have been completed since that time.

25.1.2 2016 Fenelon Deposit Mineral Resource Estimate

In 2016, InnovExplo was mandated to prepare a mineral resource estimate on the Fenelon deposit and a supporting Technical Report in accordance with National Instrument 43-101 ("NI 43-101") and Form 43-101F1 (Richard et al., 2016). A model was generated for the entire drilled area of the Fenelon deposit based on all available geological information and analytical results.

The 2016 Fenelon Deposit Mineral Resource Estimate (the "2016 MRE") was prepared by Pierre-Luc Richard, P.Geo., and Catherine Jalbert, P.Geo., using all available information.

The 2016 resource area measures 500 metres along strike, 210 metres wide and 280 metres deep. The resource estimate was based on a compilation of historical and recent diamond drill holes and wireframed mineralized zones, largely inspired by previous work and Wallbridge's interpretation. The final model was constructed by InnovExplo. In order to conduct accurate resource modelling of the deposit, the mineralized-zone wireframe model was based on the drill hole database and the authors' knowledge of the Fenelon deposit and similar deposits. InnovExplo created a total of nine (9) mineralized solids (coded 102 to 110) that honour the drill hole database.

Given the density of the processed data, the search ellipse criteria, the drill hole density and the specific interpolation parameters, InnovExplo classified the 2016 MRE as Measured, Indicated and Inferred resources. The estimate is compliant with CIM standards and guidelines for reporting mineral resources and reserves.

Following a detailed review of all pertinent information and after completing the 2016 MRE, InnovExplo concluded the following:

- Geological and grade continuity have been demonstrated for eight (8) of the nine (9) mineralized zones composing the Fenelon deposit. The ninth zone was not attributed to any resource.
- Using a cut-off grade of 5.00 g/t Au, the Measured Resources stand at 30,100 tonnes grading 13.12 g/t Au for 12,700 ounces of gold, the Indicated Resources stand at 61,000 tonnes grading 12.89 g/t Au for 25,300 ounces of gold, and Inferred Resources stand at 6,500 tonnes grading 9.15 g/t Au for 1,900 ounces of gold.
- It is likely that additional diamond drilling would upgrade some of the Inferred Resources to Indicated Resources.
- It is likely that additional diamond drilling would identify additional resources down-plunge and in the vicinity of known mineralization.

25.1.3 Exploration Potential – 2016 Technical Report

Following a detailed review of all pertinent information and after completing the 2016 MRE, InnovExplo concluded the following in the 2016 Technical Report (Richard et al., 2016):

- Geological and grade continuity have been demonstrated for eight (8) gold-bearing zones on the Fenelon Mine Property;
- A large proportion of the resource is located in close proximity to existing underground workings at shallow depth;
- The bulk of the resource is located in the first 150 metres from surface (87% of the tonnes and 91% of the ounces);
- It is likely that additional diamond drilling would upgrade some of the Inferred Resources to Indicated Resources;
- There is the potential for upgrading some of the Indicated Resources to Measured Resources through detailed geological mapping, infill drilling and systematic channel sampling from the underground workings;

- A zone that was intercepted by four mineralized intervals (Zone 110) was modelled but not interpolated, and is considered as an exploration target which requires tighter drill spacing before it can be interpolated;
- There are several opportunities to add additional resources by drilling the depth extensions of the ore shoot that originates in the resource area and the subparallel mineralized zones in the vicinity of the currently identified zones; and
- A property-scale compilation and target generation program should be completed. Conversion drilling should be devoted to upgrading part of the Inferred resources to the Indicated category, whereas the objective of exploration drilling should be to target the currently identified ore shoots at depth and discover additional zones over the entire project.

25.1.4 2017 Pre-feasibility Study

25.1.4.1 Mining Plan and Mineral Reserves

The reserves for the underground design have been estimated in accordance with the definitions and guidelines adopted by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM Standards on Mineral Resources and Reserves) in August 2000.

The mineral reserve estimate for the Fenelon deposit is based on the resource block model provided to Wallbridge by InnovExplo, along with information in the InnovExplo report titled “Technical Report and Mineral Resource Estimate for the Fenelon Mine Property”, dated August 17, 2016 (Richard and al. 2016).

The underground mine design for the Fenelon deposit will provide for a 13-month mine life, from initial underground mine rehabilitation to completion of mining. The mining plan was developed using the Indicated and Measured Mineral Resources estimate provided by InnovExplo. The Proven and Probable Mineral Reserves within the underground mine include 96,721 tonnes of ore at an average grade of 9.3 g/t Au after dilution and mining recovery factors are applied.

Prior to the commencement of mining activities, the existing open pit and underground workings should be dewatered according to local regulations and the workings would need to be rehabilitated to allow the new development to start. The mining activities would be spread over a total of six (6) levels, from which two (2) are currently partially developed and would require rehabilitation. The remaining 4 levels would need to be developed.

The main mine design parameters used in the preparation of the mine design are presented in table 25.1.

Table 25.1 – Mine design parameters – Fenelon Deposit

Subject	Units	Parameters
Mining Methods		Longhole / Uppers / Drift & Fill
Cut off Grade (CoG)	g/t	5.0
Value of the Ore at CoG	$/t (CAD)	247.00
Stope Mining Rate (target)	t/d	400
Ramp Development	m	4.5 H x 4.0 W
Level Access	m	4.0 x 4.0
Sill Development	m	3.0 x 3.0
Raise Development	m	2.44 x 2.44
Minimum Stope Width	m	2.5
Maximum Stope Width	m	12.0
Maximum Stope Height	m	18.0
Maximum Stope Strike	m	30.0
Minimum Pillar Required Between Stopes	m	5.0
Production Hole Diameter	in	2-1/4
Lateral Advance Rate	m/d	6.0
Vertical Advance Rate	m/d	2.4

The stope wireframes were manually designed using the parameters discussed above. Dilution and mining recovery factors were therefore applied on a stope-by-stope basis for each mining method. The mining plan was developed using rockfill as a backfilling method. Three rib pillar stopes have been identified and would require the use of a consolidated backfill.

Based on the nature and geometry of the Fenelon deposit, three different mining methods should be employed: long hole, uppers, and drift and fill.

Table 25.2 presents a summary of the Fenelon deposit mineral reserves and Table 25.3 lists the mine plan metrics.

Table 25.2 – Mineral Reserves Summary – Fenelon Deposit

Category	Mined Tonnes	Diluted and Recovered Tonnes	Grams Recovered	Ounces
Proven	6,321	6,770	62,970	2,025
Probable	83,974	89,951	836,600	26,897
Total	**90,295**	**96,721**	**899,570**	**28,922**

Table 25.3 – Mining metrics

Mining metrics							Total
Type		Q4 2017	Q1 2018	Q2 2018	Q3 2018	Q4 2018	
Ore production (DMT)							
	Ore	1,789	30,997	29,513	29,208	5,216	96,721
	Total	**1,789**	**30,997**	**29,513**	**29,208**	**5,216**	**96,721**
Horizontal development (m)							
	Drift - Level Access	130	227	-	-		357
	Drift and Fill Dev	-	33	58	35		127
	Sill - Rock	184	294	246	-		724
	Sill - Ore	51	220	157	-		428
	Ramp	232	260	-	-		491
	Total	**597**	**1,035**	**460**	**35**		**2,127**
Vertical development (m)							
	Raise Development	-	50	-	-		50
	Total	**-**	**50**	**-**	**-**		**50**

25.1.4.2 Metallurgy

The Fenelon ore responds well to conventional gold leaching with gold recoveries of up to 98–99% in the limited laboratory testwork done to date. The two previously tested bulk samples returned gold recoveries of 97% and 93.5%, although operational problems were reported on both occasions. The problems related to liquid losses at the gold precipitation stage and not to gold dissolution. This situation could normally be avoided by process adjustment. Taking into account the excessive gold liquid losses that occurred during the second bulk processing, both campaigns returned similar gold recoveries, with extraction close to 97%.

The additional testwork done in 2016 on historical core samples from the Fenelon deposit failed to confirm the gold recoveries by direct cyanidation. However, intensive leaching on the leach tails from those tests returned similar high gold recoveries of up to 98–99% at the target grind size. Considering the results to date on the bulk and laboratory samples, a gold recovery of 97% appears appropriate at this stage. However, it will be safe to proceed with confirmation testwork when new samples become available.

In the future, it will be critical to control liquid losses during ore processing; otherwise, the final gold recovery will be negatively affected as it was during the processing of the 2004 bulk sample. During both bulk sampling campaigns, high-pressure events occurred, but only had an obvious negative impact on the 2004 sample. Attention and proper control of gold precipitation conditions will be needed to prevent copper and/or other metals from precipitating in the precipitate press buildup.

It may be appropriate to track copper grades and optimize ore mixing to control the copper grade and sulphide variations in the mill feed. The amount of free gold suggest that gravity recovery may help control potential gold losses or trapping in the circuit during the ore processing.

The CIL gold recovery process may be a viable alternative to the current Merrill Crowe process. The Merrill Crowe, CIL or CIP should normally provide relative equivalent gold recovery. However, no CIL or CIP testwork has been done to date. In the current situation in the Abitibi area, there exist some possibilities for competitive quotations from different processing facilities. Another process plant destination will be considered as long within a positive result in term of global economic (processing cost versus additional gold recovered).

25.1.4.3 Environment

25.1.4.3.1 Environmental Considerations

The available information for the Fenelon Mine Property does not reveal any critical element that could seriously affect the future development of the project. Additional studies will have to be conducted in order to complete an environmental baseline.

Typically, mining projects have the potential of affecting their surrounding environments. With careful planning, these potential effects can usually be mitigated to render the Project acceptable to regulatory agencies.

25.1.4.3.2 Social considerations

A consultation plan will be developed to assess the perceptions of the Project by Cree, Algonquin and Jamesian communities, and to identify appropriate mitigation measures. Invitation letters were sent in November 2016 to the Algonquin and Cree communities on which the Project is located. Community representatives were invited to meet with Wallbridge to initiate dialogue and to determine how best to hold consultations with community members, or if other First Nations communities should be included.

Meetings held in the past were mostly about employability and the hiring of specialized firms and contractors with which the Crees have partnerships. Environmental quality was also mentioned as a priority issue.

25.1.4.3.3 Environmental permitting

To move forward with the Project, Wallbridge is required to submit an EIA for the Project to the Review Committee (COMEX). No formal EIA is currently needed under federal regulations.

Certificate of authorization requests will need to be submitted, which will include an analysis of potential impacts. Moreover, federal and provincial laws and regulations also govern the obligation of obtaining permits, licences or authorizations.

The EIA process is currently underway and began with the submission of the Project's preliminary information to the Evaluating Committee (COMEV) in November 2016. A directive should be issued early in 2017.

25.1.4.3.4 Ore, waste rock and water management

According to previous documents, the waste rock was considered not potentially acid generating with only a low leachability in Cd and Ba. However, when compared to current criteria, the waste rock is leachable in Ba, Cd, Cu, Mn, Ni and Zn.

As for the ore, previous documents indicate the results of all samples submitted to the static acid generation potential tests fall in the uncertainty zone. The ore is also leachable in Cd, Cu and Mn.

According to geological data, ore rocks are associated with silicification and the most abundant sulphides would be pyrrhotite (trace to 30%) and pyrite. Since pyrrhotite is the most reactive sulphide capable of causing acid mine drainage (AMD), it is recommended that the geochemical characterization be enhanced.

Ore and waste rock will be managed at surface on their respective pads. However, most of the waste rock should be used as underground filling. Underground water will be pumped in the polishing pond before being discharged to the environment. No critical quality problems were recorded for the effluent at the time of the 2004 bulk sampling.

25.1.4.3.5 Mine closure

A conceptual closure plan will have to be prepared with respect to the "*Guide de préparation de réaménagement et de restauration des sites miniers au Québec*" published in 2016. It will outline measures to be taken for progressive rehabilitation, closure rehabilitation and post-closure monitoring. It will also help refine the evaluation of restoration costs completed as part of this Report.

The conceptual plan has to be presented to the MERN for approval before the beginning of the mining activities.

25.1.4.4 Capital and operating costs

The construction and operational strategy for the mining project on the Fenelon Mine Property relies on the use of contractors. As opposed to larger projects, which are often broken into sharply defined construction and operations phases, the proposed strategy for Fenelon is to hire contractors at the start of the Project and continue the collaboration until the end of the mine life. This is made possible due to the nature, duration and scale of the Project. The project cost estimate was developed considering that contracts would be required for on-site activities, such as initial site preparation and settling pond construction, dewatering, mine development and production, surface and underground construction, ore crushing, surface buildings and camp construction as well as for operation.

It is not planned to build ore processing facilities on site; instead, the ore would be trucked to an existing facility outside the property. This strategy was used when developing the cost estimate.

The estimated pre-tax capital and operating expenditures are summarized in the following table, distributed by quarter and grouped under the main categories.

Table 25.4 – Cost expenditure summary for the mining project on the Fenelon Mine Property (C$ '000)

Cost Item	Q1 2017	Q2 2017	Q3 2017	Q4 2017	Q1 2018	Q2 2018	Q3 2018	Q4 2018	Total
Pre-production	300	668	80	0	0	0	0	448	1,496
Capital costs	0	103	1,588	1,766	1,728	53	0	0	5,238
Operating costs	0	105	1,120	3,124	7,838	5,845	4,982	697	23,710
Remote camp operations	0	0	678	400	537	537	450	438	3,041
General and Administrative	0	0	299	433	567	702	567	299	2,866
Contingency	29	66	376	560	1,049	705	593	239	3,616
Royalties	0	0	0	8	248	289	234	31	809
Total	329	941	4,140	6,292	11,968	8,131	6,825	2,152	40,777

The estimate includes contingency, which represents 9.9% of the total cost before contingency. This percentage was determined by evaluating the quantity and cost precision of each system element of the cost estimate. As a result, contingency by item varies between 5.6% and 50%. The cost estimate accuracy falls within -3% to +18%.

25.1.4.5 Financial analysis

Based on the current assumptions, discounted cash flow modelling of the project yields a pre-tax NPV of C$5.84 million at a 5% discount and a pre-tax internal rate of return (“IRR”) estimate of 92%. The NPV and IRR, after income taxes and before any withholding tax, are C$2.80 million and 60%, respectively. A summary of the results is presented in the table below.

Table 25.5 – Base case estimated financial results

Pre-tax	
NPV at 5% Discount Rate (C$ '000)	5,842
Internal Rate of Return (IRR)	92%
Payback Period	Q3 2018
After-tax	
LOM NPV at 5% Discount Rate (C$ '000)	2,802
Internal Rate of Return (IRR)	60%
Payback Period	Q2 2018

25.1.5 Risks and Opportunities

Table 25.6 identifies the significant internal risks, potential impacts and possible risk mitigation measures that could affect the future economic outcome of the project on the Fenelon Mine Property. The list does not include the external risks that apply to all mining projects (e.g., changes in metal prices, exchange rates, availability of investment capital, change in government regulations, etc.). Significant opportunities that could improve the economics, timing and permitting are identified in Table 25.7. Further information and study is required before these opportunities can be included in the project economics.

Table 25.6 – Risks for the Fenelon mining project

Expertise	Risk	Potential Impact	Possible Risk Mitigation
Metallurgy	Metallurgical recoveries are based on limited testwork	Recovery might differ negatively from what is currently being assumed	Conduct additional metallurgical tests
Metallurgy	Operational problems occurred during the two bulk sample processing campaigns	Gold was affected negatively in the second bulk sample by 3.5% (97% to 93.5%.) The effect on the first sample was not clear.	Operational problems could occur again in the future. Attention will need to be taken regarding the gold recovery process to understand the source of the problem and find a solution. CIL may prove to be a more viable process than Merrill Crowe.
Metallurgy	Ore samples used in the last characterization test (2016) were old, providing mixed results	May not be representative in terms of quality. Gold kinetics were very slow. It was not determined whether this was due to the state of the sample or another property.	Additional testwork will need to be done when new samples or ore become available.
Mining	Conditions of the existing underground mine are unknown.	The rehabilitation process may be a financial risk as well as a scheduling risk	Monitoring during the early stages of rehabilitation.
Mining	Slope conditions of the open pit while it is being dewatered are not yet known.	Rehabilitation of the slopes may be required	Monitoring during the early stages of dewatering.
Mining	Mine dewatering requirements during the operations phase have not been fully quantified	Mine water volumes may exceed calculations, leading to greater than expected demands on water management and the dewatering system.	Upgrade the existing surface water pond. Use the existing pit bottom for water management. Monitoring during the early stages of the operation and modify the dewatering system if required.

Expertise	Risk	Potential Impact	Possible Risk Mitigation
Mining	Selbaie road maintenance costs	Several users currently share the costs of Selbaie road maintenance. There is a risk that some users would no longer use this road, resulting in higher maintenance costs for the road leading to the Fenelon mine	NA
Environment	Insufficient or incomplete environmental studies or baseline data	Field work to comply with new guidelines. Higher CAPEX cost. Delay of the EIA submission, and thus the mine schedule.	Studies and field work should be performed during early stage of the EIA process.
Environment	Project located within a priority sector for the creation of a protected area for woodland caribou.	Longer analysis by the ministry, and thus a delay in the mine schedule	Early discussion with the ministry on possible mitigation measures.
Environment	Waste rock acid-generating and metal leaching	Higher CAPEX and OPEX cost for management. Higher cost for post-rehabilitation monitoring	No waste rock piled on surface
Environment	First Nations and/or social issues	Delay of the Project's social acceptance, and thus a delay in the mine schedule.	Hold meetings with stakeholders early during project development to address major issues and elaborate mitigation measures

Table 25.7 – Opportunities for the Fenelon mining project

Expertise	Opportunities	Explanation	Potential benefit
Geology	**Exploration potential**	Potential for additional discoveries at depth and around the Fenelon deposit by drilling Additional resources may be present in the immediate vicinity of the mine workings as demonstrated by the recent re-sampling program Additional resources identified by the delineation drilling of exploration targets 109 and 110, and by following zone extensions at depth	Potential to increase resources and extend mine life

Expertise	Opportunities	Explanation	Potential benefit
Metallurgy	**Metallurgy**	Recovery might be better than what is currently being assumed	Potential to increase resources and improve the viability of the project
Metallurgy	**CIL process**	CIL could be an alternative process to avoid the liquid losses occurring in the Merrill Crowe process	Potential elimination of the operational problem and stable/better gold recovery
Metallurgy	**Gravity gold recovery**	Coarse gold recovery by gravity could potentially be a good process for this ore	Minimize potential gold losses and trapping in mill
Mining	**Cost and schedule**	Early receipt of dewatering permit.	The mine could be dewatered sooner, therefore yielding a better understanding of site conditions.
Mining	**Cost and schedule**	Dewatering of the underground mine can be done via the ramp or the existing raise	Dewatering process could be accelerated.
Environment	**EIA**	Use theoretical and/or existing data to complete environmental studies.	Lower CAPEX cost. Shorter delay in submitting the EIA, thus shorter delays in the mine schedule.
Environment	**Mine Closure**	Keep and re-use surface infrastructure for use during future exploration at the end of the LOM. Use the waste rock entirely as rock fill material in open stopes	Lower mine closure cost.
Financial	**Financial**	The Company is an exploration company and has the right to use loss carry forwards	The loss carry forwards can be applied to reduce income taxes. This has not been considered in the financial evaluation of the project

25.2 Conclusions

InnovExplo, SNC-Lavalin and WSP conclude that the 2017 Pre-feasibility Study presented herein allows the project on the Fenelon Mine Property to advance to the production stage for which potential viability has been demonstrated.

26 RECOMMENDATIONS

Based on the results of this Pre-feasibility Study, InnovExplo, SNC-Lavalin and WSP recommend advancing the project on the Fenelon Mine Property to the production stage.

26.1 Site exploration and development

In order to extend the mine life and the project's financial benefits, it is recommended that above and underground exploration drilling and development work be conducted from surface and from the existing ramp. Moreover, additional exploration work will allow underground conditions to be assessed, thereby refining the mine design and lowering the risks for mining operations. By doing so, dewatering could be executed earlier in the project schedule.

While exploration work is going on, complementary engineering and environmental studies could be completed simultaneously. This will help characterize the project and site conditions, and yield a more accurate impact assessment.

The environmental impact assessment and review procedure can be conducted during exploration work. However, the more confirmed details from complementary engineering and environmental studies and a more complete mine development, operation and closure description could facilitate and accelerate a ministry review and approval of the project.

The following sections detail the recommended two-phase work program:

- Phase 1 – Exploration work and complementary engineering and environmental studies; and
- Phase 2 – Mine development and operation.

26.2 Phase 1 – Exploration Work and complementary Engineering and Environmental studies

26.2.1 Exploration work

InnovExplo recommends that Wallbridge continue to revise the property-scale compilation and to generate targets. Additional drilling should target the down-plunge extensions of the currently identified mineralized zones as described in this Technical Report. An additional objective would be the discovery of additional zones of similar mineralization near the currently identified mineralized zones. If additional work proves to have a positive impact on the project, the current resource estimate should be updated.

In summary, InnovExplo recommends the following work program:

Phase 1a:

- Initiate a property-scale compilation and target generation program;
- Conduct infill and down-plunge exploration drilling aimed at expanding the current resources.

Phase 1b (after mine dewatering and contingent upon the success of Phase 1):

- Follow-up underground drilling program on the Fenelon deposit to potentially add resources;
- Update the 3D model and resource estimate.

InnovExplo has prepared a cost estimate for the recommended work program to serve as a guideline for the Fenelon Mine Property. The budget for the proposed program is presented in Table 26.1, in section 26.4. Phase 1b is contingent upon the success of Phase 1a.

26.2.2 Environment

The following two sections present the recommended additional environmental work to obtain the required permits and to define the waste rock, ore and water management systems for the Fenelon Mine Project. These studies should be carried out between April and July 2017.

26.2.2.1 Baseline information

Additional environmental and social activities will be required to better assess the impacts of the project to be reported in the EIA. A preliminary list of these complementary activities is presented below. This list could be adjusted to meet the COMEV's directive requirements.

- Hydrological study, water surface quality, sediment quality;
- Hydrogeological study and underground water quality;
- Soil quality assessment;
- Air dispersion model;
- Waste rock and ore geochemical characterization; and
- Consultations with First Nations and stakeholders.

The environmental permitting costs, including the EIA, should be revised at the next stage of project development, as the extent of the requirements will be known following the COMEV's directive.

26.2.2.2 Ore, waste rock and water management

The next stage of geochemical characterization studies should be undertaken together with a refined assessment of waste rock quantity by lithology in order to support waste rock management options. Additional sampling and testing (static and kinetic tests) should be carried out.

A review of the surface water management infrastructure should be completed and the design should be updated during the next stage of engineering. A monitoring plan of the final effluent should be included in the surface water management plan.

The cost estimate for the environmental work is included in Table 26.1 in section 26.4

26.2.3 Metallurgy

Because additional drilling is recommended, metallurgical testwork is also recommended to confirm the gold recovery for the current and additional resources. The suggestion is to proceed with CIL testwork as an alternative to the Merrill Crowe process.

The cost estimate for the recommended work is presented in Table 26.1 in section 26.4.

26.2.4 Complementary Engineering Studies

Based on the results of this Pre-feasibility Study, it is recommended that the following work plan to be completed before commencing mining operations.

The following engineering studies should be completed before commencing mining operations:

- It is recommended that further geomechanical studies be done prior to commercial operation because the proposed bulk / longhole mining approach differs from Golder's more selective / cut & fill mining method. This study should include the final selection of the technology to be used for cemented fill.
- To design the polishing pond, it is recommended that more information be obtained on site topography, soil parameters, the existing polishing pond design (and performance) and water characteristics. This additional information can be used to update the design parameters presented in this document, if necessary. A site hydrology study is also recommended.
- At this point, no hydrogeological modelling was performed as part of the pre-feasibility study to quantify the dewatering requirements during operations. A hydrogeological model could be used to estimate the expected inflow during operations.

The estimated costs for the recommended work program are presented in Table 26.1 in section 26.4.

26.3 Phase 2 – Mine Development and Operation

26.3.1 Mining and infrastructure

At the start of the mine development, it is recommended that Wallbridge put together an owner team to work closely with and monitor the progress of the contractors working on site. The owner's team should ideally be multidisciplinary, but due to the scale of the project scale, it could be limited to essential positions. At a minimum, the team should include the following:

- Mine Manager;
- Mine Geologist;
- Mine Engineer;
- Process Engineer;
- Mine Safety and Training Officer;
- Site Security;
- Core Cutting and Sampling Technician.

Occasional consulting engineering work may be required by the owner's team during mine development and operations to support the contractor and owner teams. Costs for the owner's team and engineering support are included in the global project cost estimate.

26.3.2 Contractor mobilization

The site development strategy is based on the use of contractors. The owner should finalize contracts with the mining contractor and the camp contractor before the start of the exploration phase. The mining contractor would be in charge of installing the entire infrastructure required for dewatering, mine rehabilitation and drilling, as well as the drilling services. The camp contractor would be in charge of setting up the exploration camp.

The mill and ore transportation contracts should also be awarded to secure the milling capacity. Once the operations phase starts, the contractors would already be on site. This can be leveraged as a good opportunity for the project. The contracts should be awarded based on the owner's governance rules in order to be ready for exploration when the moving forward with the decision taken by Wallbridge.

26.4 Total Cost Estimate for Additional Work

The cost estimate for Phase 1 of the recommended work (additional exploration work and complementary engineering and environmental studies) is presented in Table 26.1, for a total of C$3,780,000, including a 20% contingency.

SNC-Lavalin, WSP and InnovExplo are of the opinion that the recommended work program and proposed expenditures are appropriate and well thought out, and that the character of the Fenelon Mine Property is of sufficient merit to justify the recommended program. SNC-Lavalin, WSP and InnovExplo believe that the proposed budget reasonably reflects the type and amount of contemplated activities.

Table 26.1 – Estimated costs for the recommended work program

Phase 1 - Work Program: Exploration Work and complementary Engineering and Environmental studies	Budget	
	Description	**Cost**
Exploration		
Property-scale compilation and target generation		$25,000
Surface drilling on the Fenelon deposit (all-inclusive)	15,000 m	$1,500,000
Subtotal 1.		**$1,525,000**
Exploration		
Follow-up underground drilling on the Fenelon deposit (all inclusive)	10,000 m	$1,000,000
Update 3D model and resource estimate		$100,000
Subtotal 2.		**$1,100,000**
Environmental Studies		$200,000
Environmental permitting ($200,000, included in the cost expenditure summary of Table 21.2 at the production stage).		$200,000
Subtotal 3.		**$400,000**
Metallurgy		
Additional metallurgical test work on the current and additional resources and CIL testwork		$50,000
Subtotal 4.		**$50,000**
Complementary Engineering Studies		
Geomechanical & backfill study		$25,000
Polishing pond engineering ($50,000 included in the cost expenditure summary Table 21.2 , at the production stage)		$50,000
Hydrology study (costs included in 2a) environmental studies)		$0
Hydrogeological study (costs included in 2a) environmental studies)		$0
Subtotal 5.		**$75,000**
Subtotal Phase 1		**$3,150,000**
Contingencies	20%	$630,000
Total Phase 1 Work Program		**$3,780,000**

27 REFERENCES

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APPENDIX I – MINING RIGHTS IN THE PROVINCE OF QUÉBEC

I.1 Mining Rights in the Province of Québec

The following discussion on the mining rights in the province of Québec was largely taken from Guzon (2012) and Gagné and Masson (2013), and from the *Act to Amend the Mining Act* (“Bill 70”) assented on December 10, 2013 (National Assembly, 2013).

In the Province of Québec, mining is principally regulated by the provincial government. The Ministry of Energy and Natural Resources (*Ministère de l’Énergie et des Ressources naturelles du Québec*: MERN) is the provincial agency entrusted with the management of mineral substances in Québec. The ownership and granting of mining titles for mineral substances are primarily governed by the *Mining Act* (the “Act”) and related regulations. In Québec, land surface rights are distinct property from mining rights. Rights in or over mineral substances in Québec form part of the domain of the State (the public domain), subject to limited exceptions for privately owned mineral substances. Mining titles for mineral substances within the public domain are granted and managed by the MERN. The granting of mining rights in privately owned mineral substances is a matter of private negotiations, although certain aspects of the exploration for and mining of such mineral substances are governed by the Act. This section provides a brief overview of the most common mining rights for mineral substances within the domain of the State.

I.1.1 The Claim

A claim is the only exploration title for mineral substances (other than surface mineral substances, or petroleum, natural gas and brine) currently issued in Québec. A claim gives its holder the exclusive right to explore for such mineral substances on the land subject to the claim, but does not entitle its holder to extract mineral substances, except for sampling and in limited quantities. In order to mine mineral substances, the holder of a claim must obtain a mining lease. The electronic map designation is the most common method of acquiring new claims from the MERN whereby an applicant makes an online selection of available pre-mapped claims. In a few areas defined by the government, claims can be obtained by staking.

A claim has a term of two years, which is renewable for additional two-year periods, subject to performance of minimum exploration work on the claim and compliance with other requirements set forth by the Act. In certain circumstances, if the work carried out in respect of a claim is insufficient, or if no work has been carried out at all, it is possible for the claimholder to comply with the minimum work obligations by using work credits for exploration work conducted on adjacent parcels, or by making a payment in lieu of the required work.

Additionally, since May 6, 2015, claim holder must submit to the MERN, on the registration anniversary date of each claim, a report of the work performed on the claim in the previous year. Moreover, the amount to be paid to renew a claim at the end of its term when the minimum prescribed work has not been carried out now corresponds to twice the amount of the work required. Any excess amount spent on work during the term of a claim can only be applied to the six subsequent renewal periods (12 years in total). Holders of a mining lease or a mining concession are no longer able to apply work carried out in respect of a mining lease or mining concession to renew claims.

I.1.2 The Mining Lease

Mining leases and mining concessions are extraction (production) mining titles that give their holder the exclusive right to mine mineral substances (other than surface mineral substances, or petroleum, natural gas and brine). A mining lease is granted to the holder of one or several claims upon proof of indications that a workable deposit could be present on the area covered by such claims, and that the holder has complied with other requirements prescribed by the Act. A mining lease has an initial term of 20 years, but may be renewed for three additional periods of 10 years each. Under certain conditions, a mining lease may be renewed beyond the three statutory renewal periods.

The Act (as amended by Bill 70) states that an application for a mining lease must be accompanied by a project feasibility study, as well as a scoping and market study as regards to processing in Québec. Holders of mining leases must then produce such a scoping and market study every 20 years. Bill 70 adds, as an additional condition for granting a mining lease, the issuance of a certificate of authorization (CA) under the *Environment Quality Act*. The Minister may nevertheless grant a mining lease if the time required to obtain the CA is unreasonable. A rehabilitation and restoration plan must be approved by the Minister before any mining lease can be granted. In the case of an open pit mine, the plan must contain a backfill feasibility study. This last requirement does not apply to mines in operation as of December 10, 2013. Bill 70 sets forth that the financial guarantee to be provided by a holder of a mining lease be for an amount that corresponds to the anticipated total cost of completing the work required under the rehabilitation and restoration plan.

I.1.3 The Mining Concession

Mining concessions were issued prior to January 1, 1966. After that date, grants of mining concessions were replaced by grants of mining leases. Although similar in certain respects to mining leases, mining concessions granted broader surface and mining rights, and they are not limited in time.

A grantee must commence mining operations within five years from December 10, 2013. As is the case for a holder of a mining lease, a grantee may be required by the government, on reasonable grounds, to maximize the economic spinoffs within Québec of mining the mineral resources authorized under the concession. It must also, within three years of commencing mining operations and every 20 years thereafter, send the Minister a scoping and market study on processing in Québec.

I.1.4 Other Information

The claims, mining leases, mining concessions, exclusive leases for surface mineral substances, and the licences and leases for petroleum, natural gas and underground reservoirs obtained from the MERN may be sold, transferred, hypothecated or otherwise encumbered without the MERN's consent. However, a release from the MERN is required for a vendor or a transferee to be released from its obligations and liabilities owing to the MERN related to the mine rehabilitation and restoration plan associated with the alienated lease or mining concession. Such release can be obtained when a third party purchaser assumes those obligations as part of a property transfer. The transfers of mining titles, and the grants of hypothecs and other encumbrances in mining rights, must be recorded in the register of real and immovable mining rights maintained by the MERN and other applicable registers.

Under Bill 70, a lessee or grantee of a mining lease or a mining concession, on each anniversary date of such lease or concession, must send the Minister a report showing the quantity and value of ore extracted during the previous year, the duties paid under the *Mining Tax Act* and the overall contributions paid during same period, as well as any other information as determined by regulation.

APPENDIX II – DETAILED LIST OF MINING TITLES

Type of Mining Tiles	Title Number	NTS Sheet	Status	Area (ha)	Registration Date	Expiration Date	Holder	Royalty
CDC	2182377	32L02	Active	55.35	16 April 2009	15 April 2019	Wallbridge Mining Company Limited (100%)	1% NSR to Balmoral Resources Ltd 1% NSR to Cyprus Canada
CDC	2182381	32L02	Active	55.34	16 April 2009	15 April 2019	Wallbridge Mining Company Limited (100%)	1% NSR to Balmoral Resources Ltd 1% NSR to Cyprus Canada
CDC	2182382	32L02	Active	55.34	16 April 2009	15 April 2019	Wallbridge Mining Company Limited (100%)	1% NSR to Balmoral Resources Ltd 1% NSR to Cyprus Canada
CDC	2271651	32L02	Active	55.37	16 February 2011	5 August 2018	Wallbridge Mining Company Limited (100%)	1% NSR to Balmoral Resources Ltd 1% NSR to Cyprus Canada
CDC	2271652	32L02	Active	55.37	16 February 2011	5 August 2018	Wallbridge Mining Company Limited (100%)	1% NSR to Balmoral Resources Ltd 1% NSR to Cyprus Canada
CDC	2271653	32L02	Active	55.37	16 February 2011	5 August 2018	Wallbridge Mining Company Limited (100%)	1% NSR to Balmoral Resources Ltd 1% NSR to Cyprus Canada
CDC	2271667	32L02	Active	55.36	16 February 2011	5 August 2018	Wallbridge Mining Company Limited (100%)	1% NSR to Balmoral Resources Ltd 1% NSR to Cyprus Canada
CDC	2271679	32L02	Active	55.35	16 February 2011	5 August 2018	Wallbridge Mining Company Limited (100%)	1% NSR to Balmoral Resources Ltd 1% NSR to Cyprus Canada
CDC	2271680	32L02	Active	55.35	16 February 2011	5 August 2018	Wallbridge Mining Company Limited (100%)	1% NSR to Balmoral Resources Ltd 1% NSR to Cyprus Canada
CDC	2271689	32L02	Active	55.34	16 February 2011	5 August 2018	Wallbridge Mining Company Limited (100%)	1% NSR to Balmoral Resources Ltd 1% NSR to Cyprus Canada
CDC	2271690	32L02	Active	55.34	16 February 2011	5 August 2018	Wallbridge Mining Company Limited (100%)	1% NSR to Balmoral Resources Ltd 1% NSR to Cyprus Canada
CDC	2271691	32L02	Active	55.34	16 February 2011	5 August 2018	Wallbridge Mining Company Limited (100%)	1% NSR to Balmoral Resources Ltd 1% NSR to Cyprus Canada
CDC	2271749	32L02	Active	55.35	16 February 2011	5 August 2018	Wallbridge Mining Company Limited (100%)	1% NSR to Balmoral Resources Ltd 1% NSR to Cyprus Canada
CDC	2271783	32L02	Active	55.36	16 February 2011	5 August 2018	Wallbridge Mining Company Limited (100%)	1% NSR to Balmoral Resources Ltd 1% NSR to Cyprus Canada
CDC	2271784	32L02	Active	42.90	16 February 2011	5 August 2018	Wallbridge Mining Company Limited (100%)	1% NSR to Balmoral Resources Ltd 1% NSR to Cyprus Canada
CDC	2271785	32L02	Active	47.74	16 February 2011	5 August 2018	Wallbridge Mining Company Limited (100%)	1% NSR to Balmoral Resources Ltd 1% NSR to Cyprus Canada
CDC	2271789	32L02	Active	53.85	16 February 2011	5 August 2018	Wallbridge Mining Company Limited (100%)	1% NSR to Balmoral Resources Ltd 1% NSR to Cyprus Canada
CDC	2271790	32L02	Active	27.44	16 February 2011	5 August 2018	Wallbridge Mining Company Limited (100%)	1% NSR to Balmoral Resources Ltd 1% NSR to Cyprus Canada
CDC	2271791	32L02	Active	51.56	16 February 2011	5 August 2018	Wallbridge Mining Company Limited (100%)	1% NSR to Balmoral Resources Ltd 1% NSR to Cyprus Canada
BM	864	32L02	Active	53.35	10 April 2007	9 April 2027	Wallbridge Mining Company Limited (100%)	1% NSR to Balmoral Resources Ltd 1% NSR to Cyprus Canada
			TOTAL	**1051.77**	**ha**		**NSR = Net Smelter Return**	**NPR = Net Profit Royalty**

APPENDIX III – METALLURGICAL TESTWORK INFORMATION AND BULK SAMPLE DATA

WALLBRIDGE MINING COMPANY

CYANIDATION TESTS (FENELON ORE)

Technical Note – Revision 1

NO: T2088

For: Mr. Attila Pentek
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Date: January 9, 2017

1 INTRODUCTION

Based on previous tests performed by the *Centre de recherche minérale* (CRM) in 1997, Wallbridge Mining Company (WMC) requested a series of cyanidation tests on Fenelon ore samples. The main objective was to evaluate gold recovery obtained from the cyanidation process on the provided ore samples.

2 METHODOLOGY

2.1. Preparation

All samples were stage-crushed to 100% < 2 mm and submitted to three homogenization passes (splitter). At this stage, 150 g of each sample were set aside for metallic sieve gold analyses and 450 g of each sample were reserved to produce a composite used for the grinding curve. The grinding curve was produced on a composite sample since the amount of material was limited.

2.2. Cyanidation

Cyanidation tests were performed following the conditions stated in a report from the CRM (Table 1). The grinding time was estimated from a 4-point grinding curve (Appendix 1). Post-cyanidation particle size was estimated using a Microtrac particle size analyser. Au grade on tailings was evaluated by duplicate fire assays and Au in solution was analyzed by ICP-MS.

Table 1: Cyanidation conditions

NaCN concentration (maintained)	1 kg/t
pH	11.5 adjusted with lime
Temperature	Room temperature (20-25°C)
Aeration	With air
Targeted P85	75 µm
Solid percentage	50%
Test duration	48 hr
Sample times (Au solution)	24 hr and 48 hr

2.3. Leachwell™

Leachwell experiments were performed on cyanidation tests tailings. To insure that sufficient Au liberation was achieved, a regrind was conducted following WMC recommendations (Table 2).

Table 2: Regrind of cyanidation tailings for Leachwell™ tests

Test	Sample	Regrind
1	1130-001	3 min
2	1050-007	3 min
3	FAB-11-25	1 min
4	1050-008	1 min
5	1040-002	3 min
6	1045-001	3 min
7	FAB-11-12	4 min
8	1040-001	4 min
9	1040-001	None
10	1040-005	4 min
11	1040-005	None

Leachwell™ experiments were carried out following the manufacturer's recommendations (Table 3), while gold in tailings was evaluated by fire assay and gold in solution measured by ICP-MS.

Table 3: Leachwell test conditions

NaCN concentration (initial)	5%
NaOH	0.7%
Leachwell™ 60X	2%
Leaching time (min 1h recommended)	2 hr

3 RESULTS

Cyanidation tests results, including reagents consumption, Cu and Au mass balances are presented in Table 4. Au tailings grades for cyanidation were calculated from Leachwell™ tests results since they were found to be more consistent and reliable than those of the fire assays, which seemed to display a nugget effect (Table 5). Particle size analysis results are presented in Table 5. Detailed technical sheets are available in Appendix 2, and ICP-MS scans for liquid and solid cyanidation tailings are available in Appendix 3.

Table 4: Cyanidation tests results

Sample	Reagents consumption		Cu			Au					
	NaCN (kg/t)	Lime (kg/t)	Solution (mg/L)	Tailings (mg/kg)	Calc. feed (%)	Solution (mg/L) 24h	Solution (mg/L) 48h	Tailings* (g/t)	Analysed feed (g/t)	Calc. feed* (g/t)	Recovery (%)
1130-001	1.86	1.8	482	2940	0.342	11.7	16.2	2.88	28.9	19.1	84.9
1050-007	0.50	1.9	52.5	673	0.073	6.8	11	14.85	26.3	25.9	42.6
FAB-11-25	0.94	1.8	236	3000	0.324	5.6	9.81	0.76	6.8	10.6	92.8
1050-008	0.39	1.0	55.3	522	0.058	2.6	5.33	0.17	6.6	5.5	96.9
1040-002	0.51	1.0	42.5	377	0.042	3.6	7.55	0.49	6.2	8.0	93.9
1045-001	0.70	1.8	156	1860	0.202	14.0	20.9	2.10	24.5	23.0	90.9
FAB-11-12	0.37	2.0	39.6	579	0.062	4.5	8.93	0.53	9.1	9.5	94.4
1040-001	0.68	0.9	141	2230	0.237	14.8	20.8	12.76	35.4	33.6	62.0
1040-005	0.60	2.0	108	1370	0.148	36.7	43.7	21.87	53.7	65.6	66.6

*Calculated from Leachwell™ test results

Table 5: Cyanidation tailings Au fire assays

	Gold grade (g/t)		
	Assay 1	Assay 2	Average
1130-001	0.14	3.77	1.96
1050-007	4.32	10.3	7.31
FAB-11-25	1.09	0.36	0.73
1050-008	0.25	0.09	0.17
1040-002	1.7	0.11	0.91
1045-001	0.65	0.14	0.40
FAB-11-12	0.27	0.32	0.30
1040-001	14.6	14	14.3
1040-005	12.9	14.3	13.6

Endal recoveries were found to be lower than expected, with the maximum being obtained from sample 1050-008 (96.9%) but with results as low as 42.6, 62.0 and 66.6% for samples 1050-007, 1040-001 and 1040-005 respectively. Also, the gold leaching kinetics, presented in Figure 1, appear to be relatively slow. In fact, most of the tests could not reach a 60% recovery after 24 hr, with half of the samples below the 50% mark. This might be explained by the higher Au grade of some of the samples; the lower recoveries were observed with sample feeds of over 25 g/t (see Figure 2). The fact that the material received was stored for more than 10 years must be considered since this could have induced oxidation. Besides, the cyanide consumption was highly related to the Cu content of the samples (Figure 3)

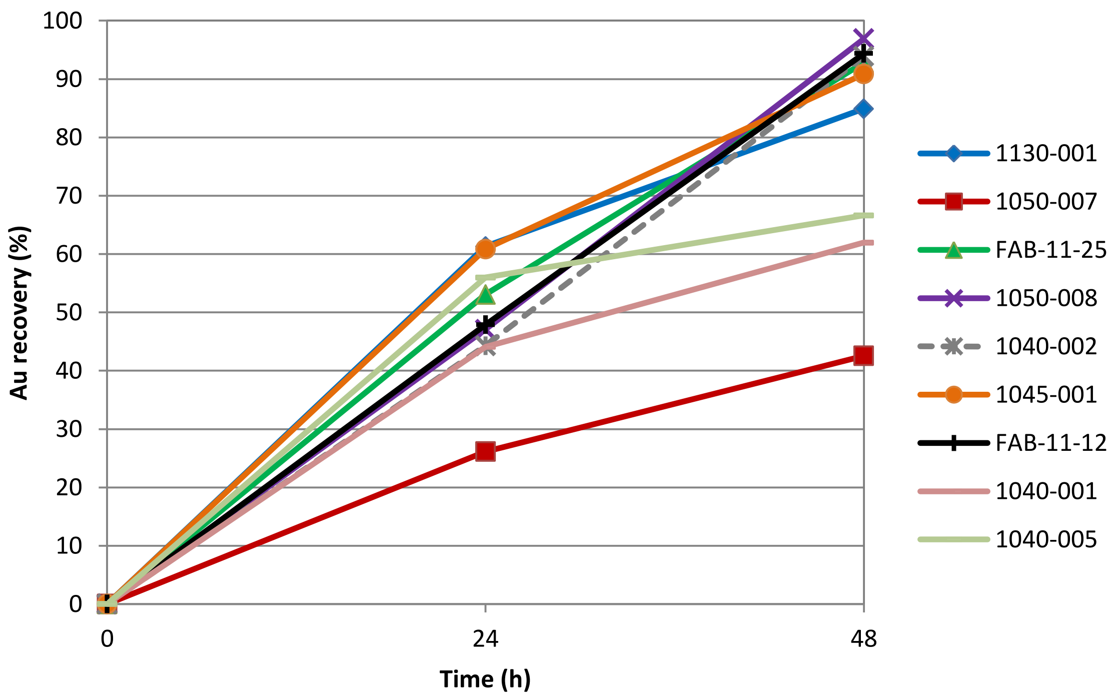


Figure 1: Gold leaching kinetics

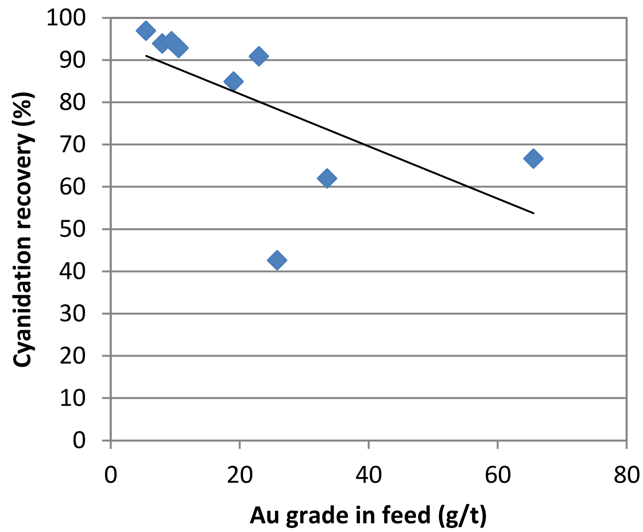


Figure 2: Cyanidation recovery compared to the feed Au grade

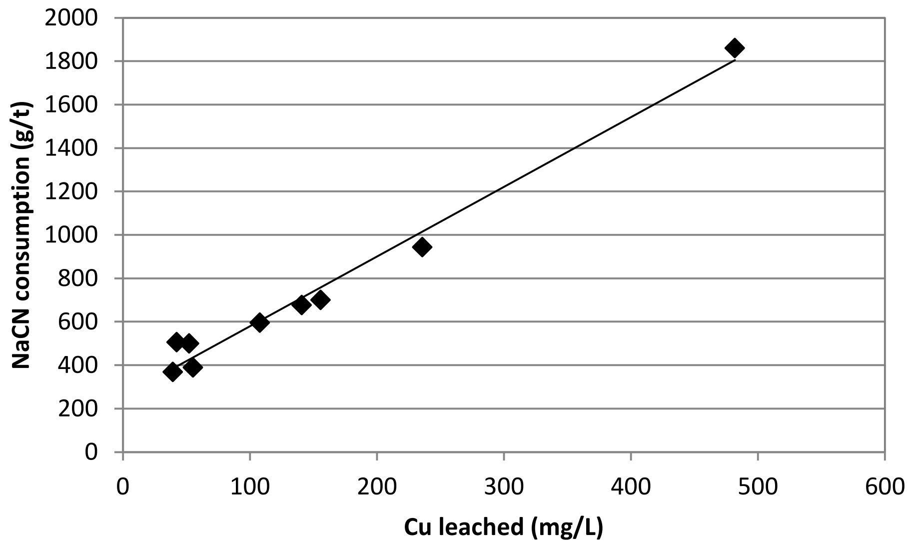


Figure 3: Effect of Cu on cyanide consumption

Table 6: Particle size analysis (Microtrac) of cyanidation tailings

Sample	Tailings P80 (µm)
1130-001	86.9
1050-007	74.9
FAB-11-25	53
1050-008	61.6
1040-002	71.6
1045-001	74.3
FAB-11-12	96.5
1040-001	89
1040-005	103

Since recoveries were lower than expected and suspecting a lack of gold exposition caused by insufficient grinding, Leachwell™ experiments were carried out on reground cyanidation tailings. Leachwell™ results are presented in Table 6 and test details are available in Appendix 4.

Tests allowed to confirm that more than 98% of the ore gold content was recoverable by cyanidation. Considering the results for samples 1040-001 and 1040-005 (with and without regrind), it seems that insufficient liberation was responsible for the loss of about 1%

overall recovery, which suggests that slow leaching kinetics seems to be the main cause of the low recoveries observed after cyanidation tests.

Table 7: Results of Leachwell™ tests on reground cyanidation tailings

Sample	Au					
	Leachwell solution (mg/L)	Leachwell solid tailings (g/t)	Calc. Leachwell feed (cyanidation tailings) (g/t)	Calc. cyanidation feed (g/t)	Leachwell recovery (%)	Overall recovery (%)
1130-001	2.5	0.38	2.88	19.1	86.8	98.0
1050-007	14.8	0.05	14.85	25.9	99.7	99.8
FAB-11-25	0.59	0.17	0.76	10.6	77.6	98.4
1050-008	0.12	0.05	0.17	5.5	70.6	99.1
1040-002	0.44	0.05	0.49	8.0	89.8	99.4
1045-001	1.97	0.13	2.1	23.0	93.8	99.4
FAB-11-12	0.47	0.06	0.53	9.5	88.7	99.4
1040-001	13.4	0.15	13.55	34.4	98.9	99.6
1040-001 (no regrind)	11.5	0.47	11.97	32.8	96.1	98.6
1040-005	19.8	0.36	20.16	63.9	98.2	99.4
1040-005 (no regrind)	22.6	0.98	23.58	67.3	95.8	98.5

4 CONCLUSIONS AND RECOMMENDATIONS

Cyanidation tests were completed and yielded relatively low recoveries after 48 hr (ranging from 42 to 96%). Leachwell™ tests were carried out to assess the amount of total leachable gold of the samples and to verify the effect of the grind size on the results. It was found that over 98% of the gold is recoverable by cyanidation and results suggest that slow kinetics are responsible for the low 48-hr cyanidation recoveries. One hypothetical cause for this assumption would be the high grade of some of the samples combined to the presence of a nugget effect.

Considering the effect of Cu on NaCN consumption, the effect of the addition of oxygen or lead nitrate, for example, could be investigated as means to improve leaching kinetics. Also, if coarse gold is present in the samples, a gravity separation step prior to cyanidation could be considered. Mineralogy analyses on cyanidation feed and tailings could provide valuable information on that matter.

APPENDIX 1:

Grinding curve

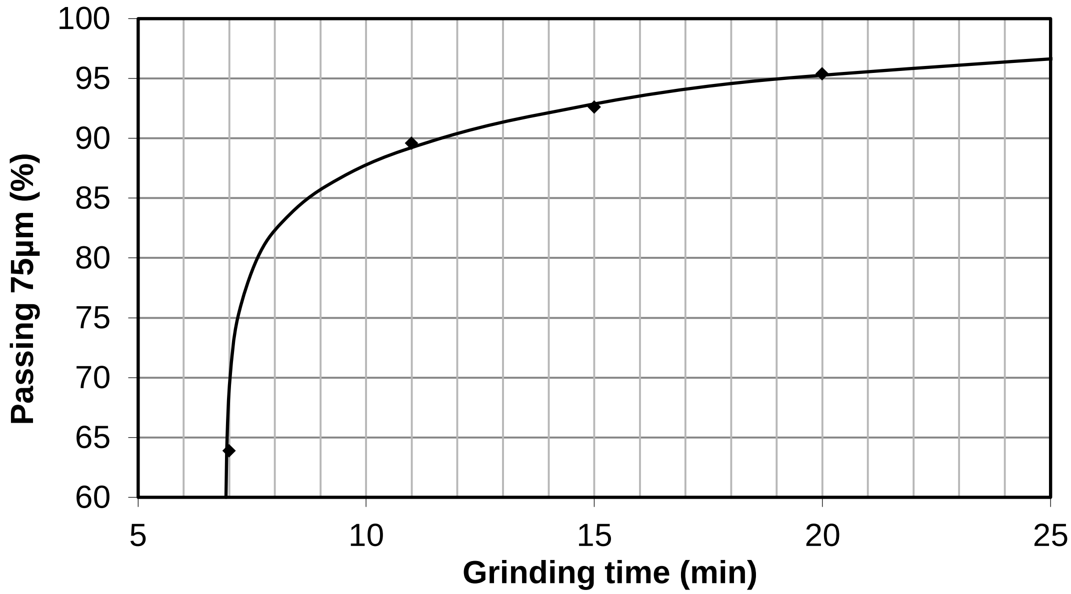


Table 8: Grinding conditions

Solid % :	50%
Ore mass (g) :	1014
Water (g) :	1014
Rod mill :	12"x6"
RPM :	50
Grinding rods	
3cm	*2873*
2cm	*1618*
1.5cm	*1201*
1cm	*721*
Total charge (g):	6413

Table 9: Particle size estimation

Grinding time (min)	**Sample mass (g)**	**Mass retained on 75µm screen (g)**	**% passing 75µm screen**
7	120.02	76.71	63.9
15	120.05	111.14	92.6
20	120.02	114.53	95.4
11 (microtrac analysis)	(-)	(-)	90

APPENDIX 2:

Cyanidation test details

Essai # 1130-001

Projet: T-2088

Date: 2016-10-31

Tech: Annie Cummings

Cyanidation parameters	
pH	11,5
NaCN (g)	1,0002
Pb(NO3) (mg)	
Temp (°C)	ambiante
Agitation (Rpm)	250
% S/L	50
Air	60

		Masse (g)
T=0h	Beaker	749,3
	Water	1018,2
	Ore	1018,09
	Reagents	1,00
	Total	2786,6
T=48h	Reactor	2601,5
	Samples taken	133,10
	Total End	2734,6

Time (h)	pH	Adjusted pH	Sample mass (g)	DO (mg/L)	AgNO3 0.005N (mL)	NaCN (mg/L)
0	8,37	11,56		8,03		
0,5	11,71			8,22		
1	11,69		27,80	7,19	3,64	447,0
3	11,59		30,90	9,15	5,94	729,0
5	11,49		29,10	9,82	6,83	838,0
24	11,51		45,30	11,25	5,15	632,0
48	11,52			11,06	4,32	530,0

Ca(OH)2 solid (g)	
Beginning	42,4077
End	41,6171
Total	0,7906
Ca(OH)2 liquid 4% (g)	
Beginning	280,5
End	254,7
Total	25,8

Automatic pH ajustment overnight

NaCN addition (h)	NaCN addition (g)	Dry sample mass (g)	
1	0,558	13,5	
3	0,2735	14,8	
5	0,1671	14,1	
24	0,3692	22,6	
48		937,01	

Grinding 11min before cyanidation

Notes:

Essai # 1050-007

Projet: T-2088

Date: 2016-11-07

Tech: Annie Cummings

Cyanidation parameters	
pH	11,5
NaCN (g)	1,0112
Pb(NO3) (mg)	
Temp (°C)	ambiante
Agitation (Rpm)	250
% S/L	50
Air	60

		Masse (g)
T=0h	Beaker	749,2
	Water	1009,5
	Ore	1009,52
	Reagents	1,01
	Total	2769,2
T=48h	Reactor	2584
	Samples taken	124,00
	Total End	2708,0

Time (h)	pH	Adjusted pH	Sample mass (g)	DO (mg/L)	AgNO3 0.005N (mL)	NaCN (mg/L)
0	8,50	11,60		10,56		
0,5	11,70			11,04		
1	11,64		31,10	11,19	5,87	720
3	11,46		18,70	11,78	7,59	931
5	11,41		22,40	11,04	6,46	793
24	11,50		51,80	10,89	8,78	1077
48	11,49			9,50	8,24	1011,0

Ca(OH)2 solid (g)	
Beginning	39,3044
End	38,6961
Total	0,6083
Ca(OH)2 liquid 4% (g)	
Beginning	282,5
End	250,6
Total	31,9

Automatic pH ajustment overnight

NaCN addition (h)	NaCN addition (g)	Dry sample mass (g)	
1	0,2918	15,6	
3	0	9,3	
5	0,2107	11,1	
24	0	25,9	
48		940,18	

Notes:

Grinding 10min before cyanidation

Airflow stuck at 20 after 48 hours

Essai # 1045-001

Projet: T-2088

Date: 2016-10-31

Tech: Annie Cummings

Cyanidation parameters	
pH	11,5
NaCN (g)	1,0113
Pb(NO3) (mg)	
Temp (°C)	ambiante
Agitation (Rpm)	250
% S/L	50
Air	60

		Masse (g)
T=0h	Beaker	733,6
	Water	1009,7
	Ore	1009,66
	Reagents	1,01
	Total	2754,0

T=48h	Reactor	2613,1
	Samples taken	102,10
	Total End	2715,2

Time (h)	pH	Adjusted pH	Sample mass (g)	DO (mg/L)	AgNO3 0.005N (mL)	NaCN (mg/L)
0	8,60	11,47		7,71		
0,5	11,54			9,19		
1	11,50			9,50	6,24	766,0
3	11,45		27,10	10,55	6,67	818,0
5	11,51		29,60	10,63	7,74	950,0
24	11,49		45,40	11,20	6,97	855,0
48	11,51			10,73	7,18	881,0

Ca(OH)2 solid (g)	
Beginning	43,1166
End	42,4915
Total	0,6251
Ca(OH)2 liquid 4% (g)	
Beginning	270,7
End	240,6
Total	30,1

Automatic pH ajustment overnight

	NaCN addition (h)	NaCN addition (g)	Dry sample mass (g)		
Notes:	1	0,2358	12,5		Grinding 10min before cyanidation
	3	0,1857	12,7		
	5	0	14,7		
	24	0,1477	22,1		
	48		939,67		

Essai # FAB-11-25

Projet: T-2088

Date: 2016-10-31

Tech: Annie Cummings

Cyanidation parameters	
pH	11,5
NaCN (g)	1,0128
Pb(NO3) (mg)	
Temp (°C)	ambiante
Agitation (Rpm)	250
% S/L	50
Air	60

		Masse (g)
T=0h	Beaker	747,9
	Water	989,4
	Ore	989,4
	Reagents	1,01
	Total	2727,7

T=48h	Reactor	2574,8
	Samples taken	102,30
	Total End	2677,1

Time (h)	pH	Adjusted pH	Sample mass (g)	DO (mg/L)	AgNO3 0.005N (mL)	NaCN (mg/L)
0	8,38	11,58		8,13		
0,5	11,74			8,75		
1	11,73		26,60	8,94	5,81	713,0
3	11,63		30,30	9,93	6,95	853,0
5	11,52			10,85	7,69	943,0
24	11,55		45,40	10,64	6,00	736,0
48	11,49			10,18	6,50	797,0

Ca(OH)2 solid (g)	
Beginning	48,4803
End	47,6876
Total	0,7927
Ca(OH)2 liquid 4% (g)	
Beginning	280
End	254,6
Total	25,4

Automatic pH ajustment overnight

NaCN addition (h)	NaCN addition (g)	Dry sample mass (g)	
1	0,2870	13,2	
3	0,1491	14,8	
5	0	14,6	
24	0,2689	22,3	
48		915,26	

Notes: Grinding 10min before cyanidation

Essai # 1050-008

Projet: T-2088

Date: 2016-11-08

Tech: Annie Cummings

Cyanidation parameters	
pH	11,5
NaCN (g)	1,0391
Pb(NO3) (mg)	
Temp (°C)	ambiante
Agitation (Rpm)	250
% S/L	50
Air	60

		Masse (g)
T=0h	Beaker	526,9
	Water	1031,3
	Ore	1031,3
	Reagents	1,04
	Total	2590,5

T=48h	Reactor	2375,95
	Samples taken	113,79
	Total End	2489,7

Time (h)	pH	Adjusted pH	Sample mass (g)	DO (mg/L)	AgNO3 0.005N (mL)	NaCN (mg/L)
0	9,13	11,52		9,46		
0,5	11,36			8,99		
1	11,51		23,70	9,92	6,17	757
3	11,46	11,58	22,89	9,99	7,49	919
5	11,48		22,83	8,18	7,54	925
24	11,15	11,65	44,37	9,35	7,04	864
48	11,03			10,40	8,14	999,0

Ca(OH)2 solid (g)	
Beginning	43,1719
End	42,1638
Total	1,0081
Ca(OH)2 liquid 4% (g)	
Beginning	
End	
Total	0

Automatic pH ajustment overnight

	NaCN addition (h)	NaCN addition (g)	Dry sample mass (g)		
Notes:	1	0,2449	11,8		Grinding 10min before cyanidation
	3	0	11,1		
	5	0	11,4		
	24	0,144	22,5		
	48				

Essai # 1040-002

Projet: T-2088

Date: 2016-11-08

Tech: Annie Cummings

Cyanidation parameters	
pH	11,5
NaCN (g)	1,0262
Pb(NO3) (mg)	
Temp (°C)	ambiante
Agitation (Rpm)	250
% S/L	50
Air	60

		Masse (g)
T=0h	Beaker	525,4
	Water	1025,4
	Ore	1025,35
	Reagents	1,03
	Total	2577,2
T=48h	Reactor	2380,69
	Samples taken	97,70
	Total End	2478,4

Time (h)	pH	Adjusted pH	Sample mass (g)	DO (mg/L)	AgNO3 0.005N (mL)	NaCN (mg/L)
0	8,91	11,55		9,17		
0,5	11,31	11,59		9,27		
1	11,53		22,50	9,59	5,48	672
3	11,35	11,55	22,35	8,97	7,26	891
5	11,42		22,15	9,08	7,90	969
24	10,98	11,61	30,70	9,34	7,24	888
48	11,10			10,74	8,55	1049,0

Ca(OH)2 solid (g)	
Beginning	53,7998
End	52,7267
Total	1,0731
Ca(OH)2 liquid 4% (g)	
Beginning	
End	
Total	0

Automatic pH ajustment overnight

NaCN addition (h)	NaCN addition (g)	Dry sample mass (g)	
1	0,3289	11,2	
3	0,1107	10,7	
5	0	10,7	
24	0,1176	15,8	
48			

Notes: Grinding10min before cyanidation

Essai # FAB-11-12

Projet: T-2088

Date: 2016-11-07

Tech: Annie Cummings

Cyanidation parameters	
pH	11,5
NaCN (g)	1,0054
Pb(NO3) (mg)	
Temp (°C)	ambiante
Agitation (Rpm)	250
% S/L	50
Air	60

		Masse (g)
T=0h	Beaker	733,5
	Water	1000,2
	Ore	1000,24
	Reagents	1,01
	Total	2734,9

T=48h	Reactor	2603,3
	Samples taken	91,70
	Total End	2695,0

Time (h)	pH	Adjusted pH	Sample mass (g)	DO (mg/L)	AgNO3 0.005N (mL)	NaCN (mg/L)	Ca(OH)2 solid (g)	
							Beginning	40,3881
0	8,94	11,55		10,69			End	39,8593
0,5	11,58			10,49			Total	0,5288
1	11,52		24,0	11,35	6,21	762	Ca(OH)2 liquid 4% (g)	
3	11,60			10,69	7,34	901	Beginning	267,7
5	11,43		22,5	10,96	5,91	725	End	230,3
24	11,50		45,2	11,14	9,64	1183	Total	37,4
48	11,52			9,63	10,35	1270,0		

Automatic pH ajustment overnight

NaCN addition (h)	NaCN addition (g)	Dry sample mass (g)		Notes:
				Grinding 10min before cyanidation
1	0,2391	11,6		After 48h, air input was found to be clogged
3	0,0997	11,4		
5	0,2832	17,9		
24	0	22,2		
48		930,53		

Essai # 1040-001

Projet: T-2088

Date: 2016-11-08

Tech: Annie Cummings

Cyanidation parameters	
pH	11,5
NaCN (g)	1,0266
Pb(NO3) (mg)	
Temp (°C)	ambiante
Agitation (Rpm)	250
% S/L	50
Air	60

		Masse (g)
T=0h	Beaker	520
	Water	1025,6
	Ore	1025,64
	Reagents	1,03
	Total	2572,3
T=48h	Reactor	2383,9
	Samples taken	109,12
	Total End	2493,0

Time (h)	pH	Adjusted pH	Sample mass (g)	DO (mg/L)	AgNO3 0.005N (mL)	NaCN (mg/L)
0	8,75	11,60		9,50		
0,5	11,65			9,14		
1	11,54		22,98	9,49	5,23	642
3	11,42	11,63	23,22	11,06	7,85	963
5	11,53		22,99	8,71	8,22	1009
24	10,94	11,65	39,93	9,42	6,59	809
48	10,96			10,26	7,19	882,0

Ca(OH)2 solid (g)	
Beginning	46,7903
End	45,8348
Total	0,9555
Ca(OH)2 liquid 4% (g)	
Beginning	
End	
Total	0

Automatic pH ajustment overnight

NaCN addition (h)	NaCN addition (g)	Dry sample mass (g)		
1	0,359	11,4		Grinding de 10min before cyanidation
3	0	11,4		
5	0	10,7		
24	0,1988	20,2		
48				

Notes:

Essai # 1040-005

Projet: T-2088

Date: 2016-11-07

Tech: Annie Cummings

Cyanidation parameters	
pH	11,5
NaCN (g)	1,0256
Pb(NO3) (mg)	
Temp (°C)	ambiante
Agitation (Rpm)	250
% S/L	50
Air	60

		Masse (g)
T=0h	Beaker	748
	Water	1020,9
	Ore	1020,92
	Reagents	1,03
	Total	2790,8
T=48h	Reactor	2659,4
	Samples taken	117,20
	Total End	2776,6

Time (h)	pH	Adjusted pH	Sample mass (g)	DO (mg/L)	AgNO3 0.005N (mL)	NaCN (mg/L)
0	8,65	11,47		10,16		
0,5	11,48			10,21		
1	11,43		24,3	10,77	5,84	717
3	11,51		22,5	10,52	6,88	844
5	11,43		23,2	11,06	7,67	941
24	11,49		47,2	10,79	7,48	918
48	11,46			9,47	6,94	851,0

Ca(OH)2 solid (g)	
Beginning	45,1628
End	44,5174
Total	0,6454
Ca(OH)2 liquid 4% (g)	
Beginning	279,6
End	243,9
Total	35,7

Automatic pH ajustment overnight

	NaCN addition (h)	NaCN addition (g)	Dry sample mass (g)		
					Grinding de 10min before cyanidation
Notes:	1	0,2907	11,9		Airflow stuck at 35 at 48h
	3	0,158	10,9		
	5	0	11,8		
	24	0	23,4		
	48		955,21		

APPENDIX 3:

Cyanidation tailings ICP-MS scans

ICP-MS scan on cyanidation liquid tailings:

Element	1130-001	1050-007	FAB-11-25	1050-008	1040-002	1045-001	FAB-11-12	1040-001	1040-005
Be	<0,02	<0,03	<0,02	<0,03	<0,03	<0,02	<0,03	<0,03	<0,03
Na	1100	671	850	553	648	738	666	674	592
Mg	0,02	0,05	0,02	0,29	0,32	0,04	0,1	0,36	0,03
Al	7,92	2,45	6,51	0,38	0,91	3,97	0,98	2,22	2,19
Si	3,1	6,6	2,7	10,2	7,5	4,6	11	5,4	3,7
P	<0,6	<0,8	<0,6	<0,8	<0,8	<0,6	<0,8	<0,8	<0,8
K	23,6	12,9	11,7	29,2	21,8	21,2	27,6	10,5	22,9
Ca	59,1	35,3	55,8	7,7	3,7	28,1	29,7	1,6	34,7
Sc	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002
Ti	<0,004	<0,005	<0,004	<0,005	<0,005	<0,004	<0,005	<0,005	<0,005
V	0,002	0,008	<0,002	0,05	0,049	0,007	0,029	0,017	0,01
Cr	<0,002	<0,003	<0,002	<0,003	<0,003	<0,002	<0,003	<0,003	<0,003
Mn	<0,002	<0,003	<0,002	0,004	<0,003	<0,002	<0,003	0,004	<0,003
Fe	3,6	3,55	4,03	2,91	5,83	4,88	3,03	8,84	4,16
Co	0,195	0,104	0,122	0,113	0,318	0,094	0,18	0,156	0,192
Ni	0,566	0,673	0,844	1,06	4,39	0,685	0,741	0,741	1,19
Cu	482	52,5	236	55,3	42,5	156	39,6	141	108
Zn	2,91	7,04	2,92	1,92	3,42	3,15	1,72	1,82	1,68
Ga	0,008	<0,005	0,005	<0,005	<0,005	0,006	<0,005	0,006	<0,005
Ge	<0,6	<0,8	<0,6	<0,8	<0,8	<0,6	<0,8	<0,8	<0,8
As	0,019	0,01	0,007	0,009	0,014	0,005	0,021	0,068	0,009
Se	<0,4	<0,5	<0,4	<0,5	<0,5	<0,4	<0,5	<0,5	<0,5
Sr	0,31	0,232	0,413	0,113	0,0642	0,195	0,344	0,0207	0,282
Zr	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002	<0,002
Mo	0,12	<0,05	0,022	<0,05	<0,05	0,02	<0,05	<0,05	<0,05
Ag	8,75	3,42	4,35	1,91	2,17	7,82	4,4	7,79	9,58
Cd	0,003	0,013	0,002	0,003	0,006	0,005	0,003	0,003	0,004
Sb	0,021	<0,05	0,011	<0,05	<0,05	0,007	0,058	<0,05	<0,05
Ba	0,0031	0,003	0,002	0,0034	0,0019	0,0035	0,0073	0,0016	0,0057
Tl	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2	<0,2
Pb	<0,002	0,004	<0,002	<0,003	<0,003	<0,002	<0,003	<0,003	<0,003
Bi	<0,08	<0,1	<0,08	<0,1	<0,1	<0,08	<0,1	<0,1	<0,1
Th	<0,0004	<0,0005	<0,0004	<0,0005	<0,0005	<0,0004	<0,0005	<0,0005	<0,0005

ICP-MS scan on cyanidation solid tailings:

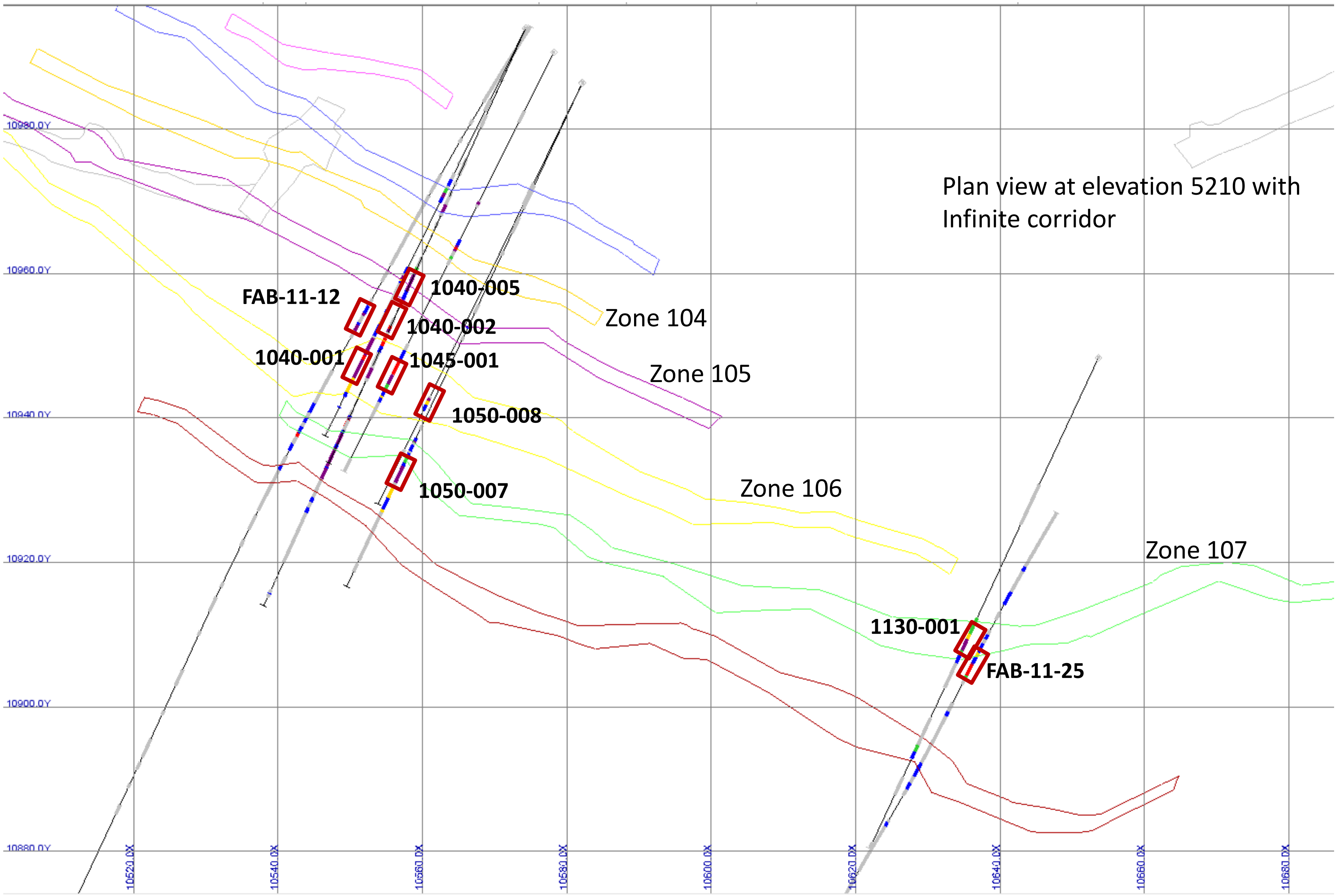
Element	1130-001	1050-007	FAB-11-25	1050-008	1040-002	1045-001	FAB-11-12	1040-001	1040-005
Be	<1	<1	<1	<1	<1	<1	1	<1	1
Na	7530	27400	18500	18800	6340	38500	12100	27900	25000
Mg	6250	26500	30900	60000	65000	17300	63100	18700	45000
Al	80500	63900	62200	58300	53900	67300	50400	59500	56000
K	39000	13000	12300	8130	6150	8830	16200	7980	11100
Ca	17000	36700	28000	62000	66700	18400	64600	29500	52900
Sc	10,9	12,7	14,1	20,2	21,6	8,7	21	8,9	15,5
Ti	1700	1120	733	2710	3400	1800	2770	1090	2930
V	71,3	115	116	145	189	75	126	69,5	135
Cr	168	360	557	1150	1370	300	1050	367	781
Mn	211	724	566	1200	1140	412	1130	796	923
Fe	35900	52300	66800	59400	67600	39600	59800	37400	68700
Co	47,8	37,5	45,1	48,6	57,9	29	48,4	26,5	60
Ni	134	161	248	396	446	158	379	148	327
Cu	2940	673	3000	522	377	1860	579	2230	1370
Zn	167	503	151	87	349	147	73	204	95
As	82	25	48	<5	34	10	56	25	37
Se	<20	<20	<20	<20	<20	<20	<20	<20	<20
Sr	129	277	214	387	356	272	278	232	455
Mo	16,8	17,1	19,3	15,5	15,6	17,4	15,6	18	19,2
Ag	4,2	0,6	3,1	0,8	<0,5	1,3	1,1	1,9	1,4
Cd	1,4	1	0,8	<0,5	0,9	<0,5	<0,5	0,9	0,8
Sb	<10	<10	13	<10	<10	<10	<10	<10	<10
Ba	617	309	293	194	104	348	463	333	310
Tl	<20	<20	<20	<20	<20	<20	<20	<20	<20
Pb	33	170	25	12	23	76	79	20	19
Bi	84	7	109	7	<5	6	52	15	13
Th	5,6	1,6	2,6	2,9	1	2,6	3	2,4	2,9

APPENDIX 4:

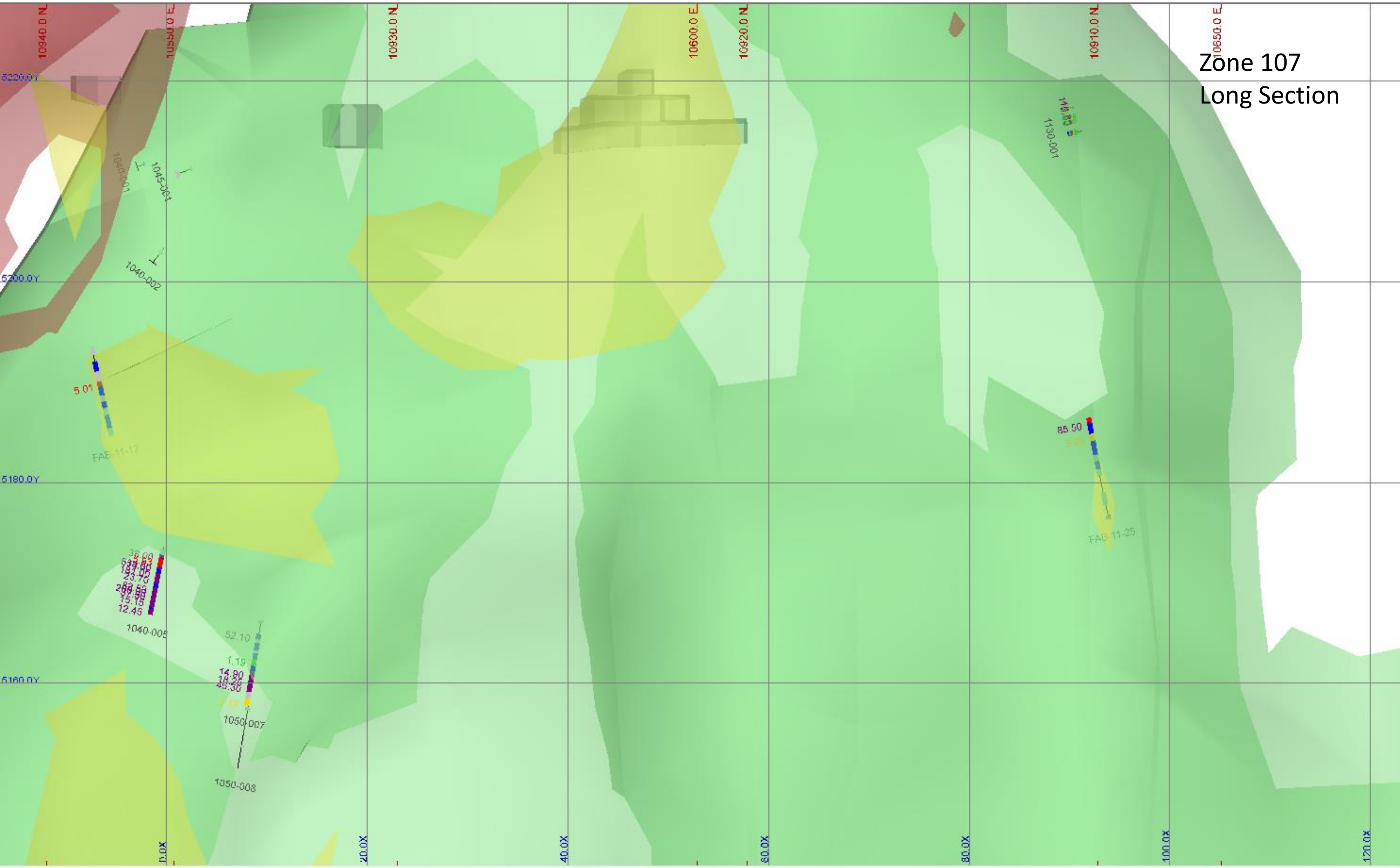
Leachwell™ test details

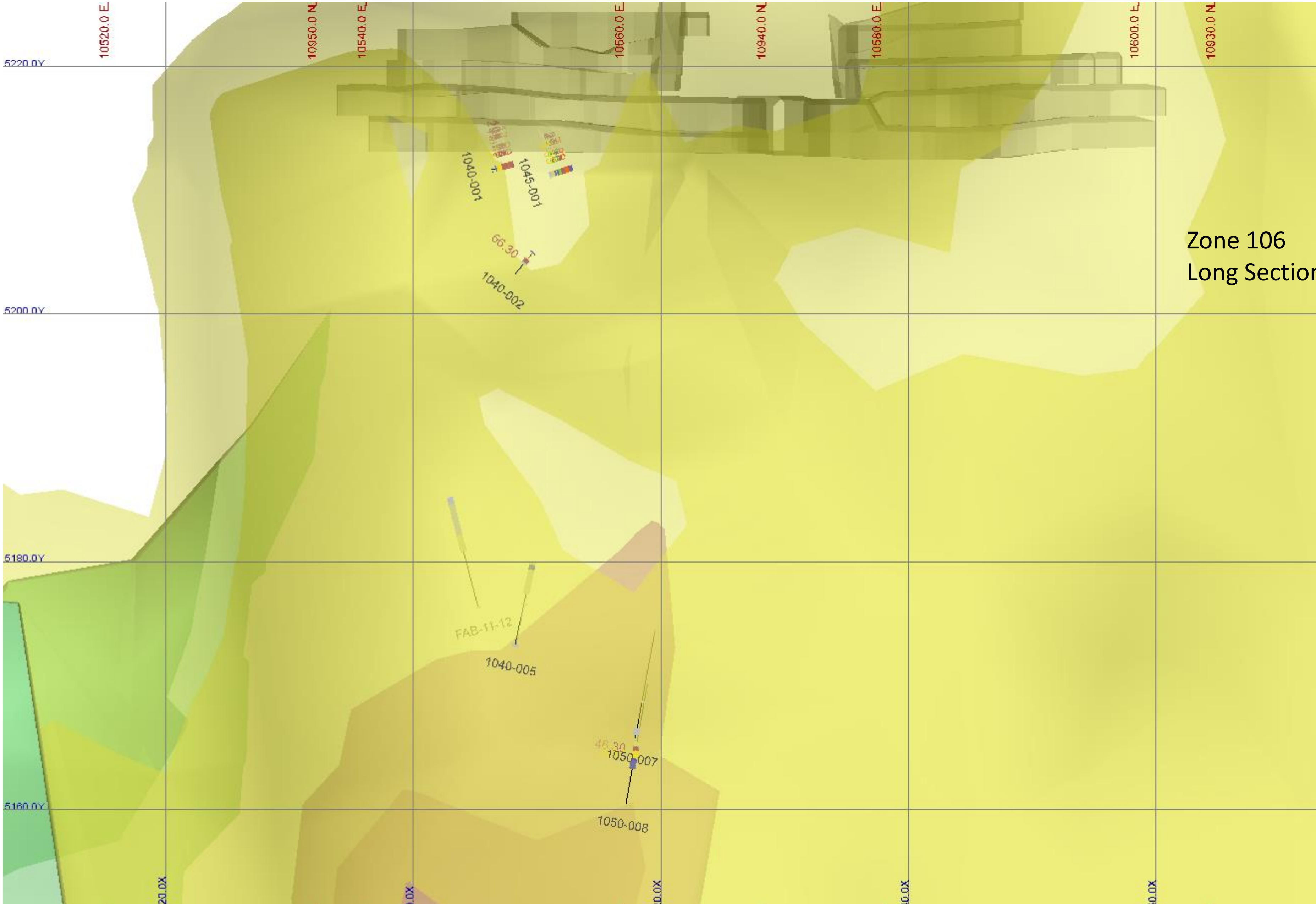
TESTS DE LIXIVIATION LEACHWELL SUR RÉSIDUS DE CYANIDATION

Test #	Sample	Initial mass g	Water g	Solid percentage %	NaOH g	LeachWELL g	NaCN g	Endal sample mass g
1	1130-001	353,64	353,74	50,0	2,5	7,0	17,53	351,59
2	1050-007	350,55	355,10	49,7	2,5	7,0	17,55	348,56
3	FAB-11-25	350,44	350,52	50,0	2,5	7,0	17,50	348,63
4	1050-008	351,95	352,07	50,0	2,5	7,0	17,50	350,06
5	1040-002	350,22	355,98	49,6	2,5	7,0	17,56	348,91
6	1045-001	350,47	350,53	50,0	2,5	7,0	17,53	349,93
7	FAB-11-12	350,31	350,39	50,0	2,5	7,0	17,52	349,95
8	1040-001	350,82	350,99	50,0	2,5	7,0	17,54	347,90
9	1040-001-NB	350,40	350,50	50,0	2,5	7,0	17,51	349,50
10	1040-005	351,72	351,71	50,0	2,5	7,0	17,61	350,01
11	1040-005-NB	350,30	350,50	50,0	2,5	7,0	17,53	348,45

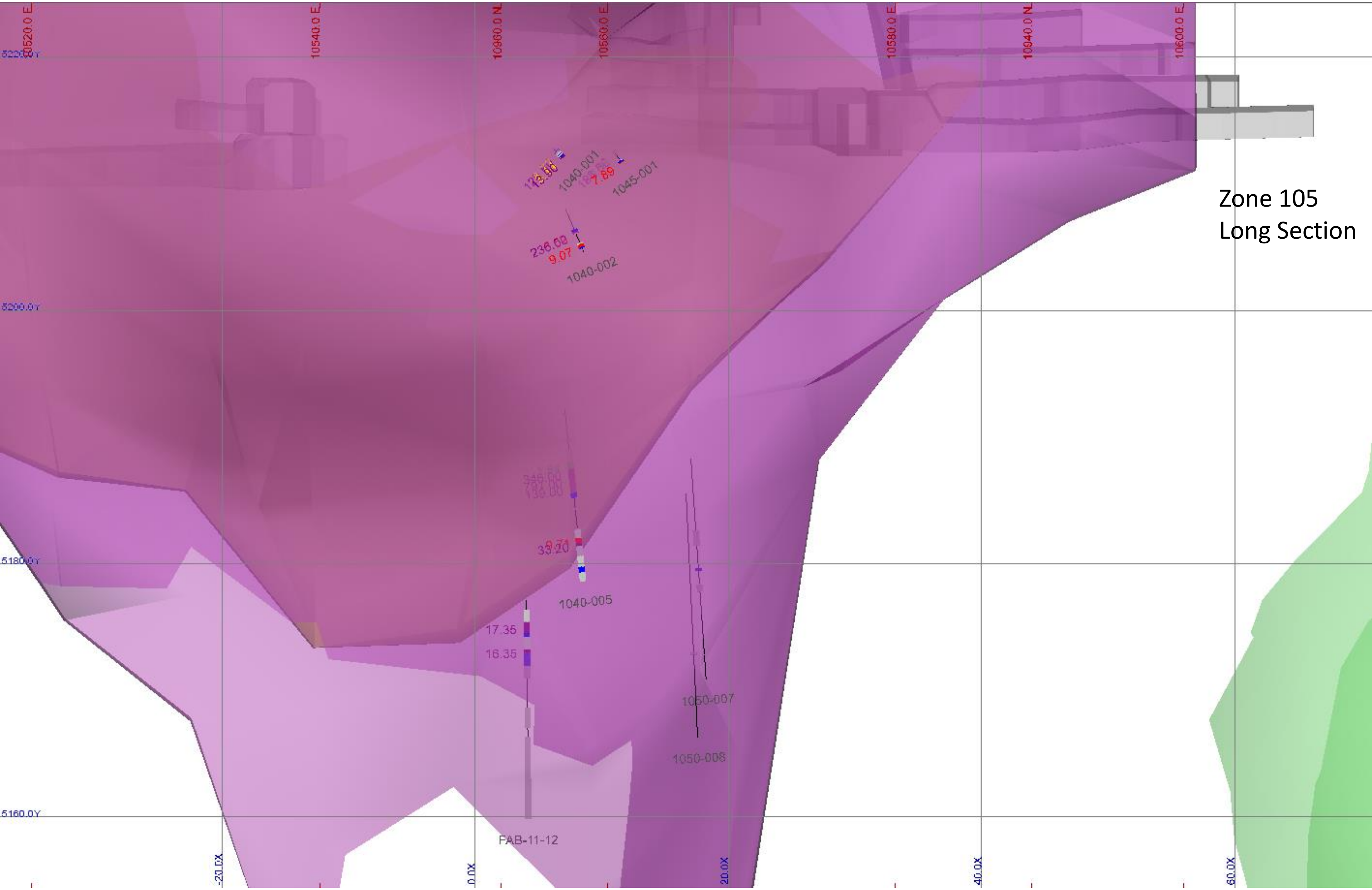

Plan view at elevation 5210 with
Infinite corridor
Zone 104
Zone 105
Zone 106
Zone 107
FAB-11-12
1040-005
1040-002
1040-001
1045-001
1050-008
1050-007
1130-001
FAB-11-25
10980.0Y
10960.0Y
10940.0Y
10920.0Y
10900.0Y
10880.0Y
10520.0X
10540.0X
10560.0X
10580.0X
10600.0X
10620.0X
10640.0X
10660.0X
10680.0X

Zone 107
Long Section

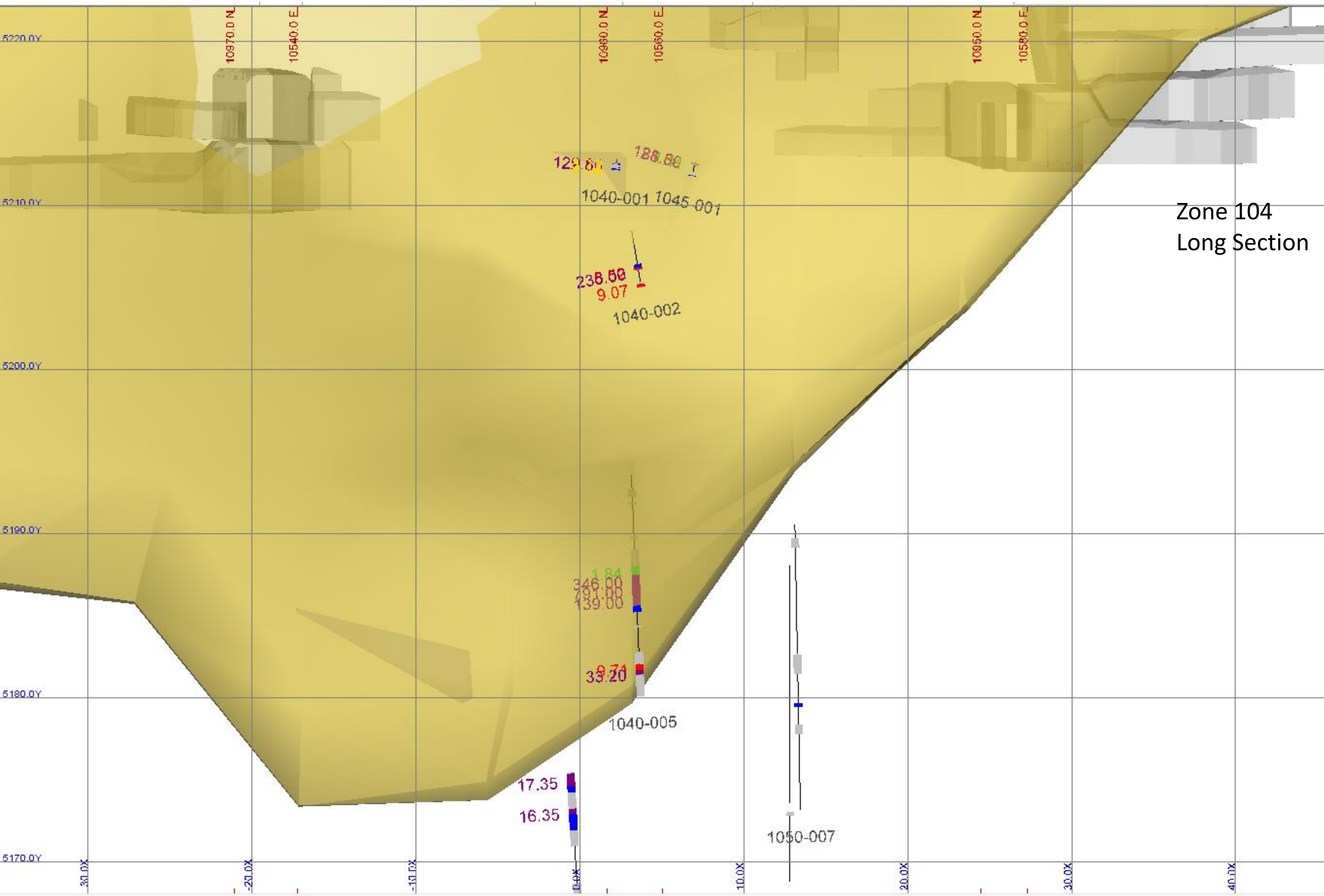




Zone 106
Long Section
10520.0 E
10950.0 N
10540.0 E
10560.0 E
10940.0 N
10580.0 E
10600.0 E
10930.0 N
5220.0Y
5200.0Y
5180.0Y
5160.0Y
20.0X
0.0X
20.0X
40.0X
60.0X
1040-001
1045-001
66.30
1040-002
FAB-11-12
1040-005
1050-007
1050-008



Zone 105
Long Section



Zone 104
Long Section
5220.0Y
5210.0Y
5200.0Y
5190.0Y
5180.0Y
5170.0Y
10970.0 N
10540.0 E
10960.0 N
10560.0 E
10950.0 N
10580.0 E
1040-001
1045-001
1040-002
238.50
9.07
1040-005
1050-007
17.35
16.35
33.20
346.00
139.00
-20.0X
0.0X
10.0X
20.0X
30.0X
40.0X

Hole ID	From	To	Zone	Au g/t		
1130-001	39.75	40.47	107	1.78		
	40.47	42.16		1.79		
	42.16	43		3.41	Met sample	41.6-44.95
	43	43.5		43.80		
	43.5	44.23		10.00		
	44.23	44.7		115.50		
	44.7	45.44		1.49		
1050-007	76	77	107	1.19	Met sample	80.00-84.00
	77	77.71		0.66		
	77.71	79.15		14.90		
	79.15	80		18.25		
	80	81		46.30		
	81	82		0.06		
	82	84		3.17		
FAB-11-25	97	98	107	0.03	Met sample	99.00-102.33
	98	99		1.01		
	99	100		5.22		
	100	101.35		7.12		
	101.35	101.65		85.50		
	101.65	102.03		0.83		
	102.03	102.33		0.75		
	102.33	103		0.83		
	103	104		3.85		
1050-008	67.17	67.58	106	0.02	Met sample	67.17-70.25
	67.58	68.16		46.30		
	68.16	69		3.87		
	69	70.25		0.62		
1040-001	47.9	50.62	106	13.90	Met sample	50.47-55.78
	50.62	51.24		8.99		
	51.24	52.02		104.00		
	52.02	52.52		13.70		
	52.52	52.99		299.00		
	52.99	54		19.85		
	54	56.33		3.72		
1045-001	46.25	48	106	0.34	Met sample	48.84-53.00
	48	49.95		7.89		
	49.95	50.04		0.07		
	50.04	50.83		27.90		
	50.83	51.43		29.10		
	51.43	52.34		1.26		
	52.34	53.23		0.61		
	53.23	53.53		3.05		
	53.53	54.19		0.10		
FAB-11-12	124.24	125.45	105	17.35	Met sample	124.24-127.98
	125.45	126		0.27		
	126	127		0.00		
	127	127.5		0.07		
	127.5	127.98		16.35		
1040-002	40.7	41.7	105	0.11	Met sample	40.7-43.5
	41.7	42		0.40		
	42	42.54		5.52		
	42.54	42.68		236.00		
	42.68	43		0.01		
1040-005	47	48.26	104	139.00	Met sample	47-50.7
	48.26	49		0.13		
	50.5	50.7		0.03		

Leach tail result

NaCN data										
	Time (h)	1130-001	1050-007	FAB-11-25	1050-008	1040-002	1045-001	FAB-11-12	1040-001	1040-005
[NaCN] (mg/L)	1	447	720	713	757	672	766	762	642	717
	3	729	931	853	919	891	818	901	963	844
	6	838	793	943	925	969	950	725	1009	941
	24	632	1077	736	864	888	855	1183	809	918
	48	530	1011	797	999	1049	881	1270	882	851
NaCN (g) Add.	1	0.558	0.292	0.287	0.245	0.329	0.236	0.239	0.359	0.291
	3	0.274	0.000	0.149	0.000	0.111	0.186	0.100	0.000	0.158
	6	0.167	0.211	0.000	0.000	0.00	0.000	0.283	0.000	0.000
	24	0.369	0.000	0.269	0.144	0.118	0.148	0.000	0.199	0.000
NaCN consumption(g/t)	1	553	280	287	243	328	234	238	358	283
	3	826	363	441	321	433	418	341	391	444
	6	994	501	506	315	466	475	619	345	505
	24	1371	433	713	376	547	570	456	545	528
	48	1860	499	943	389	506	700	369	676	595
Au leaching results										
Au leached (mg/L)	24	11.7	6.8	5.6	2.6	3.6	14.0	4.5	14.8	36.7
	48	16.2	11.0	9.8	5.3	7.6	20.9	8.9	20.8	43.7
Au solid (g/t)	48	1.96	7.31	0.725	0.17	0.91	0.40	0.30	14.3	13.6
Au Feed (g/t)	(-)	28.9	26.3	6.8	6.6	6.2	24.5	9.1	35.4	53.7
Au Feed recalculed (g/t)	(-)	18.2	18.3	10.5	5.5	8.5	21.3	9.2	35.1	57.3
Au recovery (%)	48	89.2	60.1	93.1	96.9	89.3	98.1	96.8	59.3	76.3
Bilan Cu										
Cu (liquid, mg/L)	48	482	53	236	55	43	156	40	141	108
Cu (Solid, mg/kg)	48	2940	673	3000	522	377	1860	579	2230	1370
Cu Feed recalculed (%)	(-)	0.342	0.073	0.324	0.058	0.042	0.202	0.062	0.237	0.148
Lime consumption (kg/t)	48	1.79	1.87	1.83	*0,98	*1,05	1.81	2.02	*0,93	2.03

*Lime consumption are underestimated related to pH probe problem

Detail on pyroanalyses of leaching solid tail

	Au (g/t)								
Analysis	1130-001	1050-007	FAB-11-25	1050-008	1040-002	1045-001	FAB-11-12	1040-001	1040-005
1	0.14	4.32	1.09	0.25	1.7	0.65	0.27	14.6	12.9
2	3.77	10.3	0.36	0.09	0.11	0.14	0.32	14	14.3
Average	1.96	7.31	0.725	0.17	0.91	0.40	0.30	14.3	13.6

Gouvernement du Québec
Ministère des Ressources naturelles

Sainte-Foy, le 14 octobre 1997

Monsieur Karl Glackmeyer
Les Explorations Firestar Inc.
3383, Boul. St-Jean
Suite 502
Dollard-des-Ormeaux (Québec)
H9G 3B9

Objet : Test préliminaire de valorisation du gisement Fénélon
N./Réf. : 7212 Z 098

Monsieur,

Vous trouverez ci-joints les résultats des différents travaux qui ont été réalisés sur un échantillon composé de quartz de carottes du gisement de Fénélon. Les travaux ont été démarrés le 12 septembre dernier. Vous trouverez en annexe le détail des résultats des différents travaux qui ont été effectués.

Analyse de la souche

La teneur moyenne de la souche est actuellement de 24,1 ± 2 g Au/t. Cette teneur a été déterminée à l'aide de trois analyses de l'or par la méthode des métalliques et des souches calculées pour les essais de cyanuration et l'essai de flottation.

Les analyses des autres éléments se retrouvent dans le tableau 1. Cette analyse chimique révèle qu'il n'y a pas d'éléments nuisibles à l'extraction de l'or dans la présente souche. Il faut toutefois mentionner la présence de cuivre à 0,24 % qui est un élément potentiellement consommateur de cyanure.

Séparation gravimétrique

Une séparation gravimétrique par liqueurs denses à 3,3 g/cm³ sur du minerai broyé à 48,3 % -200 mailles a été effectuée. Plus de 42,7 % demeure dans les rejets de gravimétrie. L'or n'est possiblement donc pas libre à une telle finesse de broyage.

ISO 9001
CENTRE DE RECHERCHE MINÉRALE
Direction du développement et de l'optimisation des procédés
2700, rue Einstein, Sainte-Foy, Qc, G1P 3W8 http://www.crm.gouv.qc.ca/ Téléphone : (418) 643-4540 Télécopieur : (418) 646-8106

L'étude minéralogique sur le concentré de liqueurs denses prévue ne sera pas effectuée en raison de la faible quantité d'or présente dans ce concentré. Les résultats de l'étude minéralogique ne permettraient pas de voir la majorité de l'or du gisement. L'étude minéralogique a été remplacée par la détermination du potentiel générateur d'acide des rejets de cyanuration.

Cyanuration

Pour une finesse de 85 % -200 mailles et une durée de cyanuration de 45,5 heures, la teneur des rejets de cyanuration du minerai Fénélon est d'environ 0,22 g Au/t (essais #1 et #2). Ceci correspond à une dissolution de plus de 99 % de l'or. Il convient de mentionner que le minerai Fénélon consomme peu de cyanure de sodium. La consommation de cyanure s'établit à 2,3 kg/t, avec une concentration en cyanure maintenue à 1 g/L. La consommation de la chaux s'établit pour sa part à 1,9 kg/t. Le présent échantillon se prête très bien à la cyanuration, mais possède toutefois une consommation élevée en cyanure.

Le potentiel générateur net d'acide des rejets de cyanuration a été déterminé à l'aide de la méthode développée par B.C. Research. Cette méthode permet de déterminer le potentiel d'acidification par calcul à l'aide de la teneur en soufre total. D'autre part, la capacité de neutralisation est pour sa part déterminée par titrage avec de l'acide sulfurique jusqu'à l'obtention et le maintien d'un pH de 3,5. La différence entre le potentiel d'acidification et la capacité de neutralisation détermine si l'échantillon générera de l'acide dans le parc à résidus. Si celui-ci possède un potentiel générateur d'acide compris entre -20 et +20 kg/H_2SO_4, il faudra confirmer son caractère générateur d'acide à l'aide de tests cinétiques. Ces tests cinétiques sont effectués à l'aide de micro-organismes. Le C.R.M. ne se livre pas à de tels tests, mais connaît différents fournisseurs.

Le rejet de cyanuration #1 possède un potentiel consommateur d'acide de 86,2 kg/t H_2SO_4 tandis que son potentiel générateur d'acide est de 58,1 kg/t H_2SO_4. Le potentiel générateur d'acide net est donc de -28,2 kg/t H_2SO_4. Le rejet de cyanuration de Fénélon n'est donc pas producteur d'acide. Les résultats obtenus pour le rejet de cyanuration # 2 ont confirmé les présentes observations. En effet, le rejet de cyanuration #1 possède un potentiel consommateur d'acide de 90,8 kg/t H_2SO_4 tandis que son potentiel générateur d'acide est de 58,4 kg/t H_2SO_4.. Le potentiel générateur d'acide net est donc de -32,4 kg/t H_2SO_4.

Flottation de l'or

Le rendement métallurgique de la flottation du minerai broyé à 85 % -200 mailles, est inférieur au rendement obtenu par cyanuration. Plus de 96 % de l'or a été récupéré dans le concentré de dégrossissage représentant 16,2 % du poids initial et titrant 141 g Au/t. La teneur du rejet de flottation est de 1 g Au/t alors que la teneur du rejet est de 0,22 g Au/t.

Étude minéralogique du concentré de flottation

La section polie préparée à partir du concentré de nettoyage de flottation du minerai Fénélon a été étudiée à l'aide du microscope électronique à balayage à des grossissements de 100 x dans un premier temps pour s'assurer d'observer toutes les éventuelles particules qui auraient été supérieures à 10 µm, puis à un grossissement de 1000 x pour vérifier la présence de particules aussi petites que 1 µm.

Vous trouverez ci-joints les résultats de l'étude minéralogique du minéralogiste du C.R.M., M. Jean-François Wilhelmy.

Celui-ci m'a aussi fait part des différents sulfures contenus dans le concentré de flottation, il s'agissait principalement de pyrrhotite, de chalcopyrite et de pyrite. La présence de pyrrhotite explique la consommation élevée de cyanure.

Indice de broyabilité

L'indice de broyabilité du matériel a été déterminé à l'aide de la méthode de Bond pour un broyeur à boulets en utilisant un tamis de 100 mailles (150 µm). L'indice de broyabilité se situe à 10,5 kWh/t métrique pour un broyeur à boulets. Cet indice est considéré comme celui d'une roche facile à broyer pour un minerai d'or.

Conclusions et recommandations

Le présent échantillon de minerai répond bien au traitement par cyanuration ou à la flottation. La cyanuration est toutefois préférée à la flottation pour des raisons de coûts de raffinage du concentré de flottation.

Comme l'étude minéralogique a montré essentiellement que l'or était libre et grossier. Il serait possible, voire même souhaitable, de récupérer les grains d'or les plus grossiers par gravimétrie. La gravimétrie permettrait de réduire la charge circulante d'or qui est retournée au broyeur, ce qui est d'autant plus important lorsqu'un minerai est traité à forfait.

Lorsqu'il aura plus de réserves qui auront été prouvées, il faudra procéder à une nouvelle étude métallurgique. Les conditions de cyanuration ayant trait à la finesse de broyage et à la concentration de cyanuration devront être étudiées plus attentivement. Le minerai étant consommateur de cyanure, celle-ci devrait être limitée car il s'agit d'un coût important dans le traitement d'un minerai d'or. En raison de la présence de pyrrhotite, la consommation du cyanure pourrait être limitée par une étape de préaération avec addition d'air ou d'oxygène. Lors de la préaération, il faudrait aussi évaluer la possibilité d'additionner du nitrate de plomb.

Quant à la finesse de broyage, il faudrait déterminer la finesse optimale qui permet de maximiser la récupération de l'or tout en limitant la consommation du cyanure.

Autres considérations

Veuillez noter que le C.R.M. disposera des échantillons dans 90 jours. Si vous désirez récupérer l'échantillon soumis, veuillez m'en aviser. Tel que mentionné dans l'offre de service, vous devrez assumer les frais de transport.

Nous espérons que ces résultats répondent à vos attentes et que nous aurons le plaisir de collaborer à nouveau à la réalisation de vos projets. Veuillez agréer, Monsieur, l'expression de mes sentiments les meilleurs.

Maryse St-Jean

Maryse St-Jean, ing.
Chargée de projet

p.j.
c.c.: Marc Fillion (a/s Florent Baril)

Gouvernement du Québec
Ministère des Ressources naturelles
Centre de recherche minérale

Mme. Maryse St-Jean

Résultat d'une étude minéralogique

Date :	Le 8 octobre 1997
Objet :	Nature de l'or dans un concentré de flottation
V/Réf. :	7212 Z098

Madame,

Deux prélèvement sur plaque de graphite furent effectués et étudiés en microscopie électronique à balayage (électrons rétrodiffusés) accompagnée par la spectrométrie de dispersion de l'énergie (EDX).

Au total, 15 particules d'or natif furent observées, leur diamètre allaient de 20 à 80µm. Ces particules étaient toutes libres. La répartition du matériel (volume relatif) donne les classes suivantes:

De 0 à 20 µm :	7%
De 20 à 50 µm :	8%
De 50 à 80 µm :	85%

Ces particules étaient généralement sphériques (photos # 1 à 4). Lorsque leur diamètre se maintenait autour de 50µm, on notait de nombreuses traces de remaniement apparu lors du broyage (photos # 1 et 2). Lorsque le diamètre était d'environ 20µm, les particules ne présentaient que peu de traces de remaniements et montraient même parfois les marques de leur cristallisation dans le système cubique (photos # 3 et 4) , ce qui pourrait être indicateur d'or qui aurait pu se développer en milieu ouvert (dans des zones de cisaillement sous tension). Quelques particules à la morphologie plus allongées étaient aussi présentes (photos # 5 et 6).

Jean-François Wilhelmy
Chef de projet senior
Tél. (418) 643 - 4540 ext. 247

Tableau I

Analyses chimiques

Éléments	
SiO2 (%)	57.6
Al2O3 (%)	11.2
Fe2O3 (%)	7.96
MgO (%)	5.46
CaO (%)	5.75
Na2O (%)	2.36
K2O (%)	1.44
TiO2 (%)	0.45
MnO (%)	0.11
P2O5 (%)	0.08
PAF (%)	3.59
Ag (ppm)	8.4
As (ppm)	290
C graphitique (%)	0.04
Cu (ppm)	2400
Pb (ppm)	35
Hg (ppb)	60
S (%)	2.02
Sb (ppm)	<5
Zn (ppm)	130

Densité (g/cm^3)	2.8235

Teneur en or (g/t)	
Métallique 1	24.4
Métallique 2	20.7
Métallique 3	26.2
Souche calc. cyan 1	23.5
Souche calc. cyan 2	26.2
Souche calc. flott.	23.7
Moyenne	24.1
Écart-type	2.0

ANALYSE DE L'OR PAR LA MÉTHODE DES MÉTALLIQUES

PROJET	7212Z098
TECHNICIEN(NE)	Jean-Eudes Comtois
DATE	97-09-22

Échantillon	Poids (%)	Analyses (g Au/t)				
	+150 M	+150 M	-150 M	-150 M	Moyenne -150 M	Souche
1	4.72	107.0	19.4	21.2	20.3	24.4
2	4.43	114.0	16.1	16.6	16.4	20.7
3	6.14	152.7	17.8	18.1	18.0	26.2

ESSAIS DE FLOTTATION	
PROJET	ESSAI #1
BUT DE L'ESSAI	Flottation des sulfures dans le but de concentrer l'or
ÉCHANTILLON	Fénélon
TECHNICIEN(NE)	Jean-Eudes Comtois
DATE	1997-09-

		Broyeur : #1	Charge : 17 kg
Cellule (L)	2.5	Durée (min) : 16	Finesse : 85 % -200 mailles
Vitesse (rpm)	1500	Minerai (g) : 1000	Eau (cc) : 1000

CONDITIONS

ÉTAPE	Concentration des réactifs (g/t)		Temps (min)		
	Procol CA817	MIBC	Cond.	Flot.	pH
Dégrossissage	180	9	5	3	8.7
Épuisage 1	120		1	3	8.7
Épuisage 2	100	4.5	1	3	8.6
Nettoyage 1			1	3	8.8
Ep. Nettoyage 1		4.5	1	3	8.7

Produit	Poids (%)	Analyse Au (g/t)	Distribution Au (%)
Conc. Nett.	6.70	329	93.1
Conc. Nett. Ep.	1.44	47.4	2.9
Rejet Nett.	8.06	1.28	0.4
Rejet final	83.80	1.00	3.5
Souche	100.00	23.7	100.0
Conc. Nett. + Ep	8.14	279	96.0
Conc. Dég.	16.20	141	96.5

ESSAI DE CYANURATION # 1

CONDITIONS			
Finesse: 85 % -200 M		% Solides: 50	
Cyanuration	Durée	pH	NaCN(g/L)
	~48	11.5	1.0

CONSOMMATION DES RÉACTIFS						
Durée (heures)	NaCN		CaO	$Ca(OH)_2$		
	Concentration résiduelle (g/L)	Consommation (kg/t)	Concentration résiduelle (g/L)	Consommation (kg/t)	pH avant	pH après
2.75	0.34	0.66			11.5	11.5
5	0.53	1.137			11.0	11.5
21	0.57	1.578			11.5	11.5
29	0.66	1.929			11.0	11.5
45.5	0.63	2.312	0.06	1.92	11.0	

SOLUTION			
Durée (h)	Poids (g)	Teneur Au (g/t)	Dist. Au (%)
21.0	1000.0	20.3	86.3
45.5	1000.0	23.3	99.1
REJETS			
Final	998.9	0.219	0.9
SOUCHE CALCULÉE			
Final	1000.0	23.5	100.0

La teneur des rejets a été déterminée en double, soit 0,232 gAu/t et 0,206 g Au/t.
La moyenne a été utilisée pour les fins de calculs.

ESSAI DE CYANURATION # 2

CONDITIONS			
Finesse: 85 % -200 M		% Solides: 50	
Cyanuration	Durée	pH	NaCN(g/L)
	~48	11.5	1.0

CONSOMMATION DES RÉACTIFS						
Durée	NaCN		CaO	$Ca(OH)_2$		
(heures)	Concentration résiduelle (g/L)	Consommation (kg/t)	Concentration résiduelle (g/L)	Consommation (kg/t)	pH avant	pH après
2.75	0.36	0.64			11.0	11.5
5	0.52	1.127			11.0	11.5
21	0.56	1.577			11.5	11.5
29	0.61	1.959			11.0	11.5
45.5	0.62	2.351	0.05	1.93	11.0	

SOLUTION			
Durée (h)	Poids (g)	Teneur Au (g/t)	Dist. Au (%)
21.0	1000.0	21.9	83.5
45.5	1000.0	26.0	99.1
REJETS			
Final	998.5	0.240	0.9
SOUCHE CALCULÉE			
Final	1000.0	26.2	100.0

La teneur des rejets a été déterminée en double, soit 0,229 gAu/t et 0,251 g Au/t.
La moyenne a été utilisée pour les fins de calculs.

Séparation aux liqueurs denses (48.3 % -200 mailles)

Produit	Poids (%)	Analyses Au (g/t)	Dist. Au
Concentré [1]	2.98	462.9	57.3
Rejet	97.02	10.6	42.7
Souche calculée	100.00	24.1	100.0

[1] : le concentré a été conservé en vue d'une éventuelle étude minéralogique. C'est pourquoi la teneur du concentré n'a pas été analysé, mais celle-ci a été calculé par différence à l'aide la teneur moyenne de la souche et de la teneur du rejet.

Indice de broyabilité selon la méthode Bond pour un broyeur à boulets	
PROJET	7212Z098
ÉCHANTILLON	Fénélon
TECHNCIEN(NE)	Jean-Eudes Comtois
DATE	1997-09-24

Cycle	Ajout de souche (g)	Nombre de tours	Poids de passant (g) 100 Mailles				
			Produit broyé	Ajout	Produit généré	Net/tour	Produit à broyer
1	1231.3	100	383.1	170.7	212.4	2.12	181.1
2	383.1	120	334.7	53.1	281.6	2.35	298.7
3	334.7	130	342.3	46.4	295.9	2.28	305.4
4	342.3	133	336.7	47.4	289.3	2.17	304.4
5	336.7	140	366.5	46.7	319.8	2.28	305.1
6	366.5	131	335.2	50.8	284.4	2.17	301.0

Cycle 3 à : 6 Net/tour: 2.23 Charge circulante (%): 257.18

p80 (μm) : 110.08

f80 (μm) : 2935

13.86% passant 100 mailles

Poids idéal (g) : 351.8

2.23 Net/tours

WI (kWh/t métrique) : 10.5

1231.3 g pour un volume 700 mL

DISTRIBUTION GRANULOMÉTRIQUE	
PROJET	7212Z098
TITRE	Tests préliminaires de valorisation du gisement Fénélon
BUT DE L'ESSAI	Détermination de l'indice de broyabilité
ÉCHANTILLON	Produit
TECHNCIEN(NE)	Jean-Eudes Comtois
DATE	1997-09-

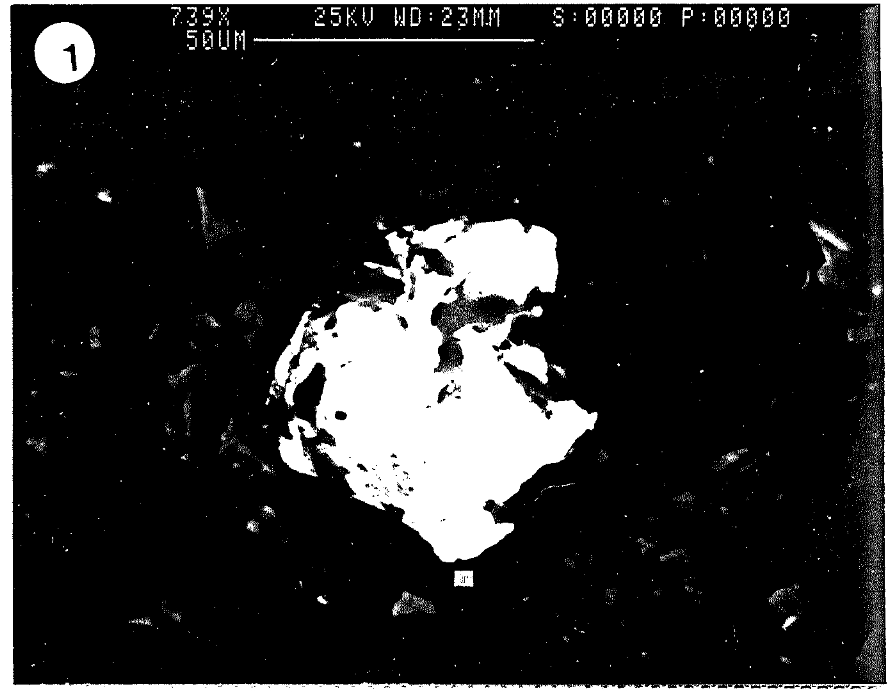
Fraction		Poids	Cumulatif	Cumulatif
mm ou µm	mailles ou po.			
Initial		(g)	retenu (%)	passant (%)
+150 µm	100	0.85	0.85	99.15
+125 µm	115	12.93	13.78	86.22
+106 µm	150	9.17	22.95	77.05
+75 µm	200	12.70	35.65	64.35
+53 µm	270	12.75	48.40	51.60
+45 µm	325	5.85	54.25	45.75
-45 µm		45.75	100.00	0.00
Total		100.00		

P_{80} (µm) = 110.1

DISTRIBUTION GRANULOMÉTRIQUE	
PROJET	7212Z098
TITRE	Tests préliminaires de valorisation du gisement Fénélon
BUT DE L'ESSAI	Détermination de l'indice de broyabilité
ÉCHANTILLON	Alimentation
TECHNCIEN(NE)	Jean-Eudes Comtois
DATE	1997-09-24

Fraction		Poids	Cumulatif	Cumulatif
mm ou μm	mailles ou po.			
Initial		(%)	retenu (%)	passant (%)
+1,70 mm	10	45.24	45.24	54.76
+1,18 mm	14	14.52	59.76	40.24
+850 μm	20	6.91	66.67	33.33
+600 μm	28	5.90	72.57	27.43
+425 μm	35	4.81	77.38	22.62
+300 μm	48	3.11	80.49	19.51
+212 μm	65	2.92	83.41	16.59
+150 μm	100	2.72	86.14	13.86
+125 μm	115	1.41	87.55	12.45
+106 μm	150	1.16	88.71	11.29
-106 μm		11.29	100.00	0.00
Total		100.00		

P_{80} (μm) = 2 935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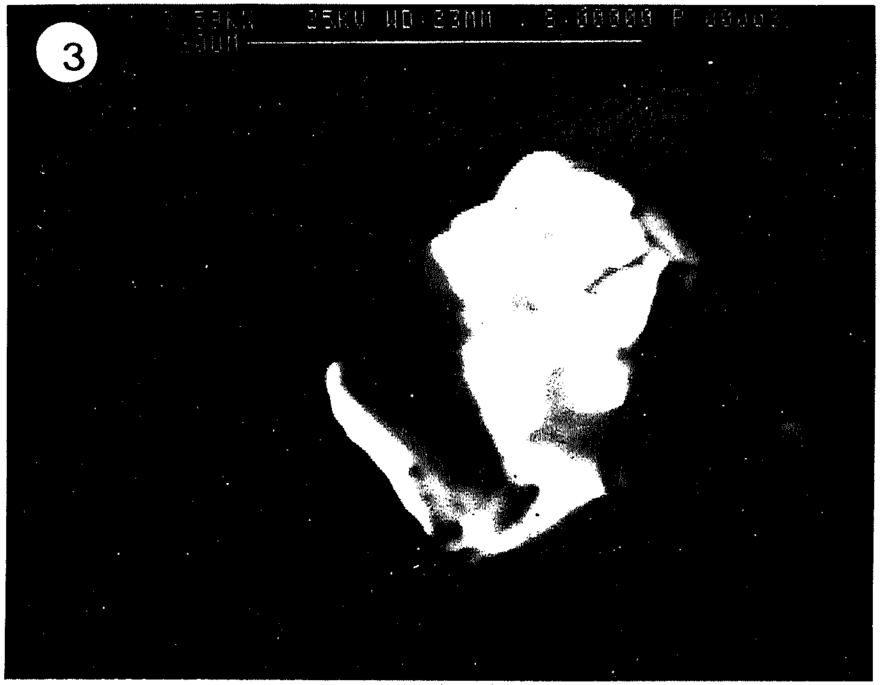
50UM


2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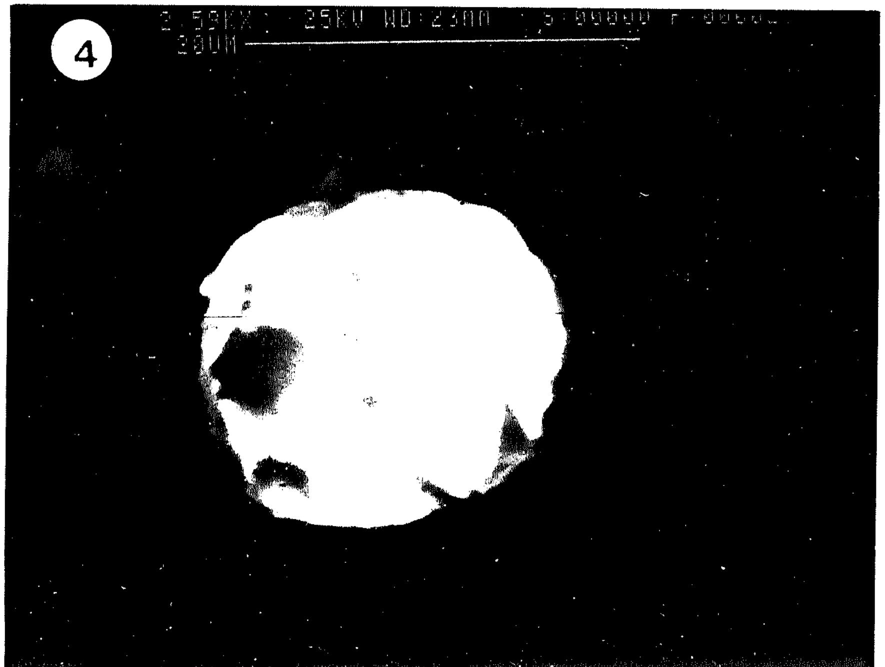
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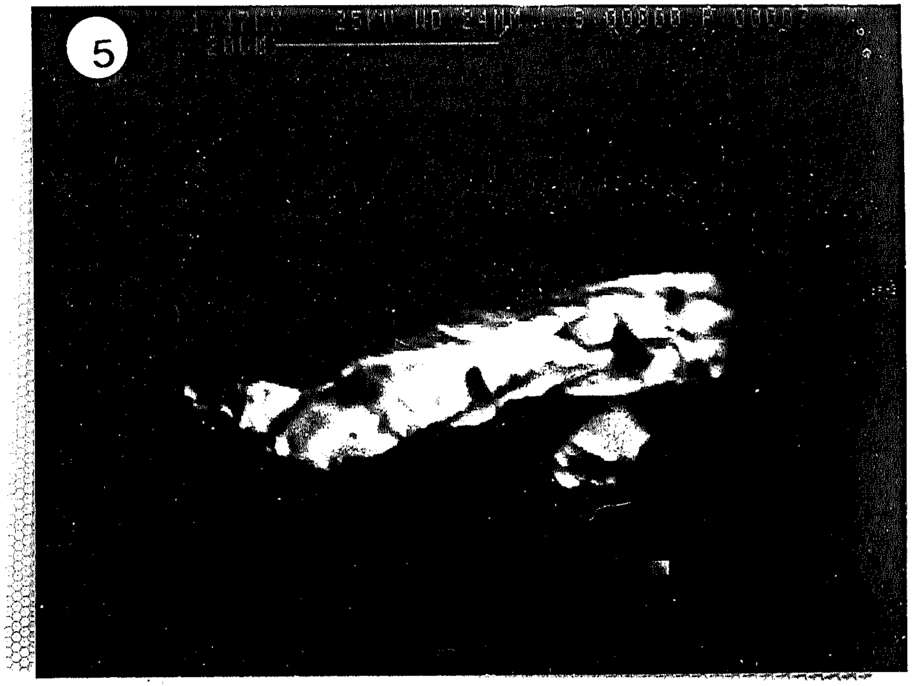
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Dubuisson, July 31, 2001

Report of Fenelon ore treatment, at Camflo milling facilities:

BRIEF HISTORY:

Two bathes of Fenelon's ore propriety were processed at the Camflo mill. The first batch from May 30 to June 04 was of 5 187 metric tonnes. The second batch from June 27 to July 06 2001 was of 8 526 metric tonnes . The total dry tonnage was of 13 713 metric tonnes with an average grade of 9,84 Au. and of 3.00 Ag. grams/tonne resulting in a recovery of 97,10%.

CAMFLO PROCESS DESCRIPTION:

Crushing Circuit:

The crushing circuit begins with a jaw crusher and a primary cone crusher in an open circuit. It is then followed by a secondary cone crusher in a close circuit to produce a final product through a ¾ x ¾ of a inches opening screen. The crushing capacity is in the range of 125 metric tonnes per hour.

Grinding Circuit:

The ore is fed at the rate of 40-45 tonnes per hour through a Rod Mill in an open circuit, the mill discharge is then mixed with the discharge of the two Ball Mills. It is then classified through a single cyclone. The underflow is used to feed both Ball Mills and the overflow is the final grinding product. The entire power consumption of the Grinding Mills is 452 kWh.

Thickening, Leaching, and Filtration:

The Cyclone overflow is feeding three similar thickeners. The thickener's underflows feeds the leaching circuit. The thickener's overflows becomes the pregnant solution. The 45 hours leaching circuit and filtration consists at the first stage of three leach tanks and two drum filters. The second circuit consist of two leach tanks and two drum filters. Finally the tailing circuit consist of one leach tank and two drum filters.

Gold Recovery:

The gold recovery is obtain by using a Merrill Crowe process, with a solution bags clarifier. Followed by a Merrill Crowe tower and Perrins presses. This produces a gold concentrate that is then melted by two fuel Wabi furnaces to produce the dory.

AVERAGE RESULTS OBTAINED:

Process:	Direct Cyanidation / Merrill Crowe
Tonnage rate:	43 metric tonnes per hour
Work index:	10.5 kWh/tm
Final grind:	± 85% minus 200 mesh and ± 65% minus 325 mesh
Head grade:	9.84 Au. grams per metric/tonne 3.00 Ag. grams per metric/tonne

Leaching time versus percent elution:

Grinding:	55.4%	Total elution:
8 Hours:	05.0%	55.9%
16 Hours:	17.5%	72.9%
24 Hours:	31.5%	86.9%
32 Hours:	36.0%	91.4%
40 Hours:	40.3%	95.7%
45 Hours:	42.4%	97.8%

Reagents consumptions kilos per metric/tonne:

Sodium Cyanide	1.00
Quick lime	1.36
Lead nitrate	0.13

Reagents consumptions kilos per metric/tonne coun't

Lead acetate	0.001
Zinc dust	0.022
Flocculent	0.015
Anti-scaling	0.025

The Metallurgical references used for Fenelon's ore treatment, is the Research (test work) done by Centre de Recherches du Québec (CRM) produced for Fairstar Exploration Inc. on September 1997.

The turn out of the milling on a commercial scale reflects exactly the CRM Research results. Except the fact that the cyanide consumption was lower by 1.3 kilos per tonne.

We ran into some minor problems. For the first batch the premature gold presses' high pressure resulted in a higher gold barren value, causing a soluble tailing lost of 0.003% higher than we expected. On the second batch an anti-scaling was injected in the clarifier tank, thus solving this abnormality. In the refining process we had to pore gold buttons prior to the bars to separate the mat from the gold. This problem should be corrected by appropriate flux and/or controlling the redox, by augmenting the bleeding of the barren during the process.

CONCLUSION:

The Camflo milling facilities were adequate to treat Fenelon's ore successfully. Fenelon's ore has a relative low work index and the presence of chalcopyrite. If the installation of some new milling facilities is required, I recommend that a S.A.G. mill and a C.I.L processing should be taken in consideration.

For any future questions do not hesitated to contact the under signed.

Annexe: Camflo flow sheet (produce for Fenelon)
Production summary

Roger Jolicoeur pdg

LABORATOIRE LTM inc.

International Taurus Resources Inc.

Milling Process – Camflo Mill

Fenelon Project

REPORT No. 1

PREPARED BY: Edmond St-Jean, P. Eng. September 2004

Table of Contents

INTRODUCTION

Mr. André Deguise, Eng., mandated us to verify the procedure followed for the milling of a lot of ore extracted from the Fenelon Project. The work required to make a first visit to discuss the sampling method, conduct a pre-milling inventory of the facilities and equipment, make random daily visits, take samples, and be present at the opening of the clarifying presses, casting of the gold bricks and cleanup of the tank house.

This report will help other individuals assigned to supervise the bulk milling of a lot of ore from the Fenelon Project to be fully documented.

1. First visit

On our first visit, it was agreed to follow the following steps:

- Measuring the quantity of ore left in the lump ore bin, and filling the latter with ore from the Fenelon Project.
- Checking the cleanliness level of the crushing circuit.
- Checking the zero-balance condition and totalizer of the scale fit to the conveyor on which ore is brought to fine ore bin # 3.
- Checking if the fine ore bin is empty (if yes, it can be filled).
- Measuring the level of pulp in the thickeners: taking pulp samples and sampling the solid/liquid interface; measuring the density of each sample; measuring the water level in the thickeners; taking surface liquid samples at various points around the thickener; taking pulp samples in the thickener underflow at the entry point of agitator no. 1. (Only solids are to be analyzed, for the water contained in the pump cap would dilute the liquid.)

- Stopping the air inlet system of the agitators and measuring the pulp level in each one of them; taking pulp samples for analysis after solid/liquid filtration; measuring the pulp density.
- Checking the pulp level in drum filters (normally these should be empty).
- Checking the level of water in the sumps (normally these should be empty).
- Checking the water level in the reservoirs that contain sterile solution, rich and re-circulated solution; taking samples.
- Sampling the ball grinders.
- Checking the totalizer of the scale that feeds the rod grinders.
- Launching the circuit using new ore.
- Changing the presses after checking the seal numbers.

2. Pre-milling Inventory

2.1 Crushing Circuit

Our visit enabled us to make the following observations:

The surface below the conveyors was very clean. So was the perimeter of the cone crushers and screens. A small quantity of ore was found on the screen recirculation conveyor.

The lump ore bin was as empty as can be. We checked the dead load level at the bottom of the bin.

We checked the zero-balance condition of the scale that brings crushed ore to the fine ore bin. We made sure that the conveyor belt was set in motion about two hours before proceeding with the zero-balance check. The error margin of the scale is ½ of 1%.

When applying the standard weight the value should be somewhere between 13,534 and 13,670.

We checked the totalizer: it read 340,606.26 short tons. We took some weight off the scale, started the auto-zero and completed 5 turns. The scale read 13,435. That value exceeded the permitted limits. We took more weight off the scale and started the auto-zero. The first value read 0.219. We started the auto-zero once again. The second value read 0.002. We repeated the operation for a third time, and got 0.002. We did it again, and obtained 0.003. The fifth and last time gave the acceptable value of 0.001.

We took more weight off the scale. We started the auto-zero, completed 5 turns, and got 13,653. That value was deemed acceptable. We repeated the operation, and obtained 13,656. We verified the totalizer after checking the zero-balance condition, and obtained 340,606.23 short tons.

Finally, we checked the fine ore bins. Bin no. 1 was 100% full of ore from the Beaufor Mine. Bin no. 2 was empty and the chutes were kept open to let particles likely to fall off the conveyor above the bin blend with the ore removed from bin no. 3. Bin no. 3 was completely empty; it is the only bin that should contain ore from the Fenelon Project.

2.2 Reservoirs

The next step consisted in checking the three thickeners. Only one thickener was being used. The following sampling procedure was followed:

- Pulp samples were taken at various levels using a wood graduated stem to which a flexible pipe was held in position with electrical insulation tape.
- Two teams were assigned to perform this operation: one at the upper level of the thickener (upper team), the other at the lower level (lower team). The

upper team started the siphon and plunged the rod into the thickener until it reached a predetermined depth. The team advised the lower team that it was ready to hold the rod in that position.

- The lower team let the pulp flow for a few minutes, then measured the pulp density and sampled it. It advised the upper team to move the rod.
- That procedure was repeated each time the rod would be moved up to pre-established levels, until it reached the clear water zone. Then the upper team drove the rod down to the interface. The level of interface was noted.
- The underflow was sampled at various points around the thickener.
- The thickener underflow was sampled, but only solid was sent for analysis, because the water contained in he pump cap would have diluted the liquid.

Next we took a sample of the pulp in the agitators. There are six of them. The air inlet system was stopped so as to take an accurate measure of the pulp level. Then we measured the pulp density and took a sample. This procedure was repeated for every agitator.

We checked the pulp level in the filter bases. All of them were empty.

We checked the level of pulp in the sumps. They were empty.

We checked the level of the rich, poor and recycled solution in the reservoirs. We took a sample in each one of them.

2.3 Grinders

We took a sample from the three grinders, took the reading of the totalizer of the scale that feeds the rod grinder, and restarted the circuit using ore from the Fenelon Project.

2.4 Presses

We changed the presses after checking the seal numbers. We put new seals on the valves that allow for press change.

3. Milling Operation

3.1 Follow-up

One hour before introducing ore from the Fenelon Project into the circuit, we had the cyanide level increased to 1.2 lb per short ton. The initial tonnage had been established at 50 short tons per hour. We had oxygen added into agitator no. 2. No problem was encountered during the first half-day (Thursday, September 9).

Friday afternoon we went back to the mill, and noted that the pressure in the presses had increased abnormally. Mr. Gérald Lavoie was advised to stop adding anti-precipitating agent in the clarifier. Mr. Lavoie said he would.

The following afternoon (Saturday), we realized that not only anti-precipitating agent had been added, but scale solvent too. The pressure had gone up. In the evening, we shook the presses by insufflating pressurized air into them. The color test showed signs of gold loss over a period of six hours during that night, but that the situation had gone back to normal. I asked the solutions operator to stop adding any of the above products. He did.

Sunday morning the pressure in the presses was still going up, though more slowly. The color test showed no sign of gold loss.

The following morning (Monday) the operation report revealed that many color tests showed signs of gold loss on Sunday evening. The pressure in the presses was rising. In the afternoon, Mr. Richard Nolet phoned me to advise that the analysis results showed signs of substantial gold loss. We decided to empty and change the presses on Tuesday morning. Mr. Nolet had the tonnage reduced to keep gold loss to a minimum.

Tuesday morning we opened the presses. I theorized that the pressure rise in the presses resulted from the formation of zinc hydroxide. Hydroxides are hard to filter. To verify my theory, I suggested that the quantity of lead salt added to zinc powder should be increased. That was done.

The presses were opened in accordance with the following procedure.

We applied maximum safety measures to empty the presses. Two security guards of the Mirado Agency attended the operation from start to end.

To access the presses every person concerned needed a specific key. The following persons were present: the two security guards, Gérald Lavoie (Operations Foreman), Edmond St-Jean, P. Eng., consultant for Fenelon, and four operators, who emptied the presses.

The seals were checked. Before cutting them, the security guards noted their respective number. A huge pan was placed underneath each press to collect the precipitate. The presses were opened, one operator standing on each side of each press, making sure that the precipitate would drop into the pans. The cotton cloth in contact with the precipitate was tucked in to make sure that the precipitate clinging to it would be preserved. The cloth was put in a container inside of which a black plastic bag had been spread out.

Each time a container would be full, a lid would be put on it and sealed twice. That operation was repeated until both presses were empty. The pans were removed from under the presses and emptied into screw-topped containers inside of which black plastic bags had been spread out. The pans were cleaned with a cloth, and put in the containers with the press cotton cloths.

All containers with precipitate and press cottons were transferred to the tank house under the supervision of the security guards and Messrs. Lavoie and St-Jean.

The operators and one security guard went back to the press cage to reassemble the presses. They put pans underneath the presses so as to recover whatever small quantity of precipitate that could come loose. At the end, they cleaned the pans with cotton cloths and wrapped them in cotton cloths. The cloths were held in position using adhesive tape.

The following afternoon (Wednesday), we noted that the pressure had hardly gone up. We also noted the presence of a large quantity of graphite in the thickeners and agitators. We sampled the graphite in the agitators, and had the tonnage increased so as the surface area exposed to graphite would be kept to a minimum. I had a discussion with Messrs. Nolet and Lavoie concerning the gold content in the poor solution. The three of us came to the following conclusion: the few hours during which gold loss occurred as reported by the solutions operators by no means justify the analysis results, which revealed that the grades were 25 times superior to normal. In order to obtain grades that high, the operators would need to note signs of gold loss on every test. That was not the case.

The following day (Thursday), we proceeded with the inventory, and at 10:45 a.m. we stopped feeding the circuit with ore from the Fenelon Project.

4. Inventory (discussion)

4.1 Inventory No. 1

Nothing particular was noted during the first inventory. The sampling method proved very efficient and ironed out. We checked for ore accumulations in the cyclone overflow under the screen collecting plastic debris from blasting. No noteworthy accumulation was observed. However the calculation grid used to prepare the inventory report is hardly professional. Volumes are expressed in cubic feet by foot. The cubic feet by foot are multiplied by kilograms per liter and after using a series of conversion ratios the products are expressed in short tons.

On checking, we found that the conversion ratio error gave a maximum error of less than 0.2%, which is still very acceptable. Since it occurred before and after the inventory, the error nullified.

4.2 Inventory No. 2

In the course of this inventory, it took several hours to empty bin No. 3, for fine particles had formed a heap that would not react to vibration. Again we checked for ore accumulation in the cyclone overflow under the screen collecting plastic debris from blasting, but no noteworthy accumulation was observed.

The inventory was conducted smoothly, and besides the fact that bin no. 3 was hard to empty, nothing special was observed.

5. Refining

The refining process was conducted under the supervision of two security guards of the Mirado Agency. The presses were emptied while two operators started the furnaces and burned the cloths used to change the presses.

After the presses were emptied, we all went to the tank house taking the plastic barrels containing the precipitate along with us. The two operators started blending the precipitate with the flux. That operation was conducted efficiently.

When casting the first brick, we had the unpleasant surprise to see a piece of refractory cement coming off the furnace shell. As a result, we had to pursue the operation using only one furnace.

Matte formed when casting the second brick, and increased as we cast the third and fourth brick. That simply resulted from our adding rich slag coming with the brick in the furnace that would produce the next brick.

Besides that, nothing particular happened during the refining process. Each brick was marked and weighed. After casting the last brick, we recovered a 921.9-gram button, and after cleaning the furnace, we recovered a 207.1-gram button. The four bricks weighed 3,427.6 troy ounces in total.

6. Tank House Cleanup

The tank house cleanup operation began Monday, September 20, at 8 a.m., under the supervision of a security guard. We weighed the poor slag, poured it in two brown barrels (# 469 and # 470) and put a seal on them.

We removed pieces of matte from the rich slag. Then we poured the rich slag into a bucket together with what had been recovered from cleaning the brick molds. The bucket was weighted, sealed, and marked *Taurus*. The rest of the rich slag was poured into buckets; these were weighted, sealed and marked *Taurus*. The buckets were kept inside the tank house.

The inner surface of the furnace hood was scraped and jet streamed so as to recover every little piece of slag. The recovered slag was then carefully put in a bucket. The latter was weighted, sealed and marked *Taurus*.

7. Feed Grades

This particular lot of ore is composed of two zones: one is rich, the other is poor. It is hard to draw a dividing line between the two of them during the operation. However Table 1 shows a variation in feed grades.

Table 1

Date	Dry tonnage st	Feed grade ounce/st	Recovery %
8	548	0.295	98.51
9	1194	0.416	98.25
10	1120	0.356	97.67
11	1186	0.378	94.27
12	1192	0.363	91.49
13A	732	0.310	93.48
13B	382	0.355	94.20
14	1018	0.188	94.19
15	1111	0.110	96.61
16	522	0.149	97.31

The above table shows that we milled about 6,354 short tons of rich ore grading some 0.362 ounce per short ton. Poor ore represents some 2,651 short tons grading approximately 0.148 ounce per ton.

8. Bond's Index

We took the opportunity to determine a dynamic Bond's Index based on the size distribution of the ore when entering and leaving the grinding circuit, taking the hourly tonnage capacity and grinder voltage/amperage into account. The following parameters were used to conduct the test:

- Voltage	2300 volts
- Amperage - Rod grinder	60 amps
- Amperage - Ball grinder No 1	83 amps
- Amperage - Ball grinder No 2	84 amps
- Hourly tonnage capacity	50 st/hr
- D80	10,000 microns
- D80	70 microns

Using the above voltage, amperage and tonnage capacity parameters, it is possible to calculate that the circuit uses 10.44 kWh/st:

2300 volt * (60+83+84)A = 522 100 Watts

Therefore 522.1 kW divided by 50 st/hr = 10,44 kWh/st

We can calculate the Bond's index (Wi) based on the above energy requirements and the distribution size (incoming/outgoing):

$$\frac{10\ Wi}{(70)^{0.5}} - \frac{10\ Wi}{(10\ 000)^{0.5}} = 10.44\ kWh/st$$

$$\frac{10\ Wi}{8.37} - \frac{10\ Wi}{100} = 10.44\ kWh/st$$

$$1.19\ Wi - 0.1\ Wi = 10.44\ kWh/st$$

Therefore Wi = 8.85

9. CONCLUSIONS

Based on the results obtained for the milling of 9,005 short tons of ore from the Fenelon Project, we drew the following conclusions:

- The Camflo Mill personnel showed professionalism during the milling operation.
- The copper contained in the ore contributed to cyanide consumption, but had very limited impact on the refining process.
- It seems that the graphite contained in the ore caused no recovery problem.

Based on the same milling results, we drew the following conclusions:

- The rich ore represents 6,354 short tons grading some 0.362 ounce/st.
- The poor ore represents some 2,651 short tons grading 0.148 ounce ounce/st.
- High pressure in the presses may have resulted from zinc hydroxide formation. It is possible to solve that problem by increasing the amount of lead added to zinc powder.
- The addition of MILL-SPERS 805 scale solvent into the clarifier has a negative impact on pressure in the presses.

Based on the same results:

- It is not possible to prove that adding oxygen in agitator no. 2 fostered gold recovery.
- It is not possible to prove that increasing the cyanide rate in the circuit had any impact but that of increasing cyanide consumption.
- It is not possible to provide an explanation for gold loss at the early stage of the milling process, for solutions operation reports indicate very limited loss. There are three possibilities: 1) The operators neglected to perform the tests; 2) The reagents were expired or inadequately measured out; and 3) The samples were contaminated during handling.
- Given the problems encountered during the process, we cannot draw conclusions concerning the effect of the distribution size on gold recovery.

10. RECOMMENDATIONS

Before milling another lot, the effect of copper on cyanidation must be verified. That involves checking the possibility to float copper before or after cyanidation, and comparing results with direct cyanidation.

Floating copper before cyanidation aims to minimize cyanide consumption and increase revenues from the sale of copper concentrates. The gold recovery rate must be checked as well, for it is possible that copper concentrates carry a significant amount of gold, of which only 95% would be paid by Noranda.

Floating copper after cyanidation aims to increase revenues from the sale of copper concentrates and augment the gold recovery rate, for that coming after the concentrate would be lost during cyanidation.

Before another lot of ore is milled, making sure it does not contain graphite is advised. If it does, the effect of graphite on gold recovery must be verified.

In the course of the next milling operation, we strongly suggest to double the quantity of lead added to zinc powder and to keep an eye on the pressure level in the presses. In absence of graphite, the milling process should be set at the lowest tonnage possible for maximum ore grinding results.

Inventory Report – September 2004
Comments

The following comments relate to inventory calculation and circuit operation.

The inventory report reveals a difference of 19.924 ounces in favor of International Taurus Resources inc. On checking the calculation, it appears that the above figure is accurate to at least the second digit after the decimal point. This minor inaccuracy is related to the method of calculation used at Camflo. Camflo rounds down the figures, on the sheet and on the screen. If recalculated, the difference would show up to the second digit after the decimal point. While the error is acceptable, their method is wrong.

The report does not take into account the amount of gold in the matte, rich slag and what was recovered after cleaning the tank house, because they were not analyzed. It is probable that they contain several ounces of gold (from 5 to 10 ounces).

It is noteworthy that the belt grade (0.299 ounce/ton) is very close to the calculated grade (0.312 ounce/ton): a difference of about 4%. That means the ore was hardly affected by the nugget effect.

The quantity of gold lost to the wastes during the milling process amounted to 77.821 for milling 9,995 short tons of ore, while 159.693 ounces were lost for 8,899 short tons. In other words, the mill malfunction resulted in the loss of about 90 ounces of gold, which would normally be recoverable. Given the tons milled, the amount of gold loss should have been 69.287 ounces.

Rapport d'inventaire de septembre 2004
Commentaire

Les commentaires de cette page sont relatifs au calcul des inventaires et du bilan d'opération du circuit.

Le bilan des inventaires donne un écart de 19,924 onces en faveur d'International Taurus Resources inc. Selon la vérification des calculs, ce chiffre est vrai au moins jusqu'au deuxième chiffre après le point. Cette légère imprécision provient du fait que Camflo arrondit dans le calcul et non juste à l'écran ce qui fait que si on refait leurs calculs on obtient une différence allant jusqu'au deuxième chiffre après le point. C'est une erreur acceptable mais c'est une mauvaise façon de faire.

Le bilan ne tient pas compte de l'or contenu dans la matte, la slag riche et le nettoyage de la raffinerie car ces produits n'ont pas été analysés. Il est probable que ces produits contiennent plusieurs onces (de 5 à 10 onces).

Il est intéressant de noter que la teneur de la courroie (0,299 once/tonne) est très près de la teneur calculée (0,312 once/tonne) soit un écart d'environ 4 %. Ceci signifie que ce minerai est peu touché par l'effet pépite.

Finalement, on peu remarquer que lors de l'usinage la quantité d'or perdu dans les rejets de l'usine était de 77,821 pour l'usinage de 9995 tonnes courtes de minerai tandis que pour le présent usinage, on a perdu 159,693 onces pour un tonnage de 8899 tonnes courtes. On peu donc dire que des mauvais fonctionnements de l'usine on fait perdre environ 90 onces d'or normalement récupérable. Pour le nombre de tonne qui a été usiné, les pertes en or auraient dû être de 69,287 onces.

Edmond St-Jean ing.

Edmond St-Jean ing.

RICHMONT MINES INC.

INTERNATIONAL TAURUS INC.

PRODUCTION

MILLING FROM SEPTEMBER 08 TO SEPTEMBER 16

TOTAL SHORT TON MILLED		8 899		
		OUNCES	GOLD RETAINED	OUNCES FOR SALE
BAR	# 007 @ 010 RCM	2 576,434 RCM	0,902	2 575,532
INVENTORY ADJUSTMENT		19,924		19,924
		2 596,358	Ounces for sale	2 595,456
GOLD BUTTON TO MR. BLAKESTAD		21,800		
TOTAL PRODUCTION		2 618,158		
FINAL GRADE		0,2942 OUNCE PER TON		
		10,090 GPT		

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TAURUS

PRODUCTION NET

PRODUCTION : TAURUS

PÉRIODE DU 08 SEPTEMBRE 2004 AU 16 SEPTEMBRE 2004

TONNAGE TOTAL USINÉE **8899**

				moins retenue	
LINGOT #	#007 @ #010	2574,685	selon lab. Bourlamaque	0	2574.685
+ AJUSTEMENT :					
ONCES SELON BILAN NET		19.924	PROPRIÉTÉ DE:	TAURUS	19.924

PRODUCTION NET	2594.609	ONCES	ONCES POUR VENTE	2594.609

ONCES REJETS	159.693
+ ESTIMÉ BOUTON	21.800
REMIS À M. BLAKESTAD	
TOTAL	2776.102

TENEUR COURROIE	0.2[illegible]	ONCE/TONNE
TENEUR CALCULÉE	0.312	ONCE/TONNE
RÉCUPÉRATION	93.46%	

Scories: L'usinage du 08 au 16 septembre 2004 a produit un total de 1,073 livres de scories. Nous estimons, selon les analyses que 5.917onces sont contenues dans les scories. Les scories sont entreposées dans les barils # 469 & #470 portant les scellés no.4411 & no.4412 et cinq (5) chaudières de nettoyage portant les scellés no.4413, #4414 et no.4415 & #4416, et no. 4417 & #4418, et no.4419 & #4420, et no. 4421 & #4422.

J'ACCEPTE, APRÈS VÉRIFICATION LE RÈGLEMENT DE 19,924 ONCES EN FAVEUR DE INTERNATIONAL TAURUS RESOURCES INC

[signature] ing.

TAURUS
BILAN NET

BILAN MÉTALURGIQUE DE L'OR NET

BILAN AVANT TAURUS

TONNAGE SOLIDE BILAN AVANT:	TAURUS	2357	TONNES
ANALYSE REJETS SOL. & LIQ.		0.0020	
ONCES REJETS INVENTAIRE AVANT:	TAURUS	6.363	
ONCES SOLIDES & LIQUIDES BRUT		158.602	
ONCES SOL. & LIQ. NET		152.239	ONCES RÉCUPÉRABLE

BILAN APRÈS TAURUS

TONNAGE TOTAL USINÉE:	TAURUS	8899			
TONNAGE SOLIDE INVENTAIRE APRÈS:	TAURUS	2456			
		6444	TONNES		
ONCES REJETS TOTALES :	TAURUS	134.626			
ONCES REJETS SOLIDES INVENTAIRE AVANT		6.363			
		128.263	ONCES	0.0199	ON/TON
ONCES REJETS SOLIDES INVENTAIRE APRÈS		31.430			
ONCES REJETS TOTALES	TAURUS	159.693	ONCES	0.0179	ON/TON
TOTAL ONCES SOLIDES & LIQUIDES BRUT		203.593			
MOINS: ONCES REJETS INVENTAIRE APRÈS		31.430			
ONCES SOL. & LIQ. NET		172.163	ONCES RÉCUPÉRABLE		

ÉCART DES BILANS NET

BILAN AVANT :	TAURUS	152.239
BILAN APRÈS :	TAURUS	172.163

CES → 19.924 ONCES SONT LA PROPRIÉTÉ DE : TAURUS

INVENTAIRE AVANT

Scellé valve fermée (camflo):	4368		
Scellé valve ouverte(client) :	4367		
Heure lecture de la balance :	12:45	Lecture :	997051.6
Lecture du débimètre :	479458		
Heure valve client ouverte :	12:47		

NOM DE LA MINE: **TAURUS**

DATE: 08 SEPTEMBRE 2004

Avant	S.G: 2.8	Mesure En Pouces	Densité En Kilo/Litre
Agitateur No 1		27.50	1.5600
2		44.00	1.5700
3		71.25	1.5650
4		44.50	1.6200
5		79.00	1.5550
6		83.00	1.6700
Solution riche		16.72	1.0000
Clarificateur			1.0000
Solution pauvre		12.50	1.0000
Solution reciculée		90.50	1.0000
Réservoir de trop plein			
Broyeur No 1			1.7000
Broyeur No 2			1.7500
Épaississeurs No 1 a			1.0000
b			1.0000
c		33.00	1.0000
d		50.00	1.0000
e		66.00	1.0100
f		99.00	1.0300
g		131.00	1.3050
Cone			1.6200
Épaississeurs No 2 a			1.0000
b			1.0000
c			1.0000
d			1.0000
e			1.0000
f			1.0000
g			1.0000
Cone			1.0000
Épaississeurs No 3 a			1.0000
b			1.0000
c		33.00	1.0000
d		66.00	1.0000
e		72.00	1.0000
f		99.00	1.0300
g		131.00	1.3250
Cone			1.5600

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AJUSTEMENT ONCES & REJETS

TAURUS

	BEAUFOR 08 SEPTEMBRE 2004	TAURUS 16 SEPTEMBRE 2004
BILAN NET ONCES	-19.924	19.924
REJETS SOLIDES	-16.895	16.895
REJETS LIQUIDES	-8.172	8.172

RICHMONT

RAPPORT

INVENTAIRE DE MINERAI

PROJET TAURUS

INVENTAIRE AVANT LE: 08 SEPTEMBRE 2004

INVENTAIRE APRÈS LE: 16 SEPTEMBRE 2004

ANALYSE DES REJETS
TAURUS

1) PREMIÈRE ESTIMATION

A) ANALYSE DES COURROIES

TAURUS

DATE	TONNES	ANALYSES	ONCES
2004-09-08	548	0.2950	161.660
2004-09-09	1194	0.4160	496.704
2004-09-10	1120	0.3560	398.720
2004-09-11	1186	0.3780	448.308
2004-09-12	1192	0.3630	432.696
2004-09-13	732	0.3100	226.920
2004-09-13A	382	0.3550	135.610
2004-09-14	1018	0.1880	191.384
2004-09-15	1111	0.1100	122.210
2004-09-16	522	0.1490	77.778
2004-09-29	-106	0.2961	-31.390
TOTAL PREMIÈRE ESTIMATION	8899	0.2990	2660.600

B) ANALYSE DES REJETS

TAURUS

DATE	TONNES SOLIDES	TONNES LIQUIDES	TONNES SOLIDES	TONNES LIQUIDES	ONCES TOTAL
2004-09-08	548	575	0.0020	0.0006	1.441
2004-09-09	1194	1298	0.0028	0.0005	3.992
2004-09-10	1120	1162	0.0056	0.0007	7.085
2004-09-11	1186	1202	0.0136	0.0053	22.501
2004-09-12	1192	1192	0.0238	0.0080	37.906
2004-09-13	732	743	0.0176	0.0080	18.827
2004-09-13A	382	382	0.0176	0.0080	9.779
2004-09-14	1018	1088	0.0156	0.0058	22.191
2004-09-15	1111	1204	0.0054	0.0033	9.972
2004-09-16	522	529	0.0030	0.0018	2.518
2004-09-29	-106	-109	0.010793	0.004055	-1.586
TOTAL REJETS	8899	9266			134.626

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C) CALCUL DES REJETS :

INVENTAIRE APRÈS

DATE	TONNES SOLIDES	TONNES LIQUIDES	TONNES SOLIDES	TONNES LIQUIDES	ONCES TOTAL	
2004-09-14	1018	1088	0.0156	0.0058	22.191	100% TAURUS
2004-09-15	1111	1204	0.0054	0.0033	9.972	100% TAURUS
2004-09-16	522	529	0.0030	0.0018	2.518	100% TAURUS
TOTAL	2651	2821	0.0088	0.0040	34.681	

TONNES EN INVENTAIRE APRÈS 2455.50

ANALYSES SOLIDES	0.0088	TONNES SOLIDES	21.608
ANALYSES LIQUIDES	0.0040	TONNES LIQUIDES	9.822
			31.430

D) CALCUL DES REJETS :

INVENTAIRE AVANT

DATE	TONNES SOLIDES	TONNES LIQUIDES	TONNES SOLIDES	TONNES LIQUIDES	ONCES TOTAL	
2004-09-06	713	693	0.0020	0.0008	1.842	100%
2004-09-07	1375	1352	0.0020	0.0003	3.156	100%
2004-09-08	1367	1344	0.0020	0.0011	4.212	100%
TOTAL	3455	3389	0.0020	0.0007	9.210	

TONNES EN INVENTAIRE AVANT 2356.63

ANALYSES SOLIDES	0.0020	TONNES SOLIDES	4.713
ANALYSES LIQUIDES	0.0007	TONNES LIQUIDES	1.650
		TOTAL REJETS INVENTAIRE AVANT	6.363

	Rejets totaux pour la période		134.626
PLUS:	REJETS POUR INVENTAIRE APRÈS	TAURUS	31.430
MOINS:	REJETS POUR INVENTAIRE AVANT	TAURUS	6.363
	Rejets totaux :	TAURUS	159.693

Inventaire total avant :	TAURUS	158.602
Rejets totaux dû à l'inventaire avant		6.363
Inventaire récupérable avant :	TAURUS	152.239
Inventaire total après :	TAURUS	203.593
Rejets totaux dû à l'inventaire après :		31.430
Inventaire récupérable après :		172.163

TAURUS
BILAN AVANT
08 SEPTEMBRE 2004

BILAN MÉTALURGIQUE DE L'OR

ECHANTILLONS	TONNES		CAMFLO TENEUR		TAURUS TENEUR		TENEUR MOYENNE		ONCES	
	SOLIDES	LIQUIDES	SOLIDES	LIQUIDES	SOLIDES	LIQUIDES	SOLIDES	LIQUIDES	SOLIDES	LIQUIDES
BROYEUR #1	6.23	3.50	0.5475	0.3400	0.5835	0.3420	0.5655	0.3410	3.523	1.194
BROYEUR #2	7.08	3.54	0.2980	0.1980	0.3160	0.1980	0.3070	0.1980	2.174	0.701
AGITATEUR #1	383.25	303.08	0.0030	0.0780	0.0040	0.0770	0.0035	0.0775	1.341	23.489
AGITATEUR #2	386.89	298.17	0.0025	0.0840	0.0020	0.0830	0.0023	0.0835	0.890	24.897
AGITATEUR #3	345.73	269.89	0.0020	0.0850	0.0020	0.0840	0.0020	0.0845	0.691	22.806
AGITATEUR #4	421.21	286.31	0.0020	0.0140	0.0020	0.0130	0.0020	0.0135	0.842	3.865
AGITATEUR #5	329.55	264.03	0.0020	0.0130	0.0020	0.0140	0.0020	0.0135	0.659	3.564
AGITATEUR #6	389.89	234.85	0.0020	0.0040	0.0020	0.0040	0.0020	0.0040	0.780	0.939
SOLUTION RICHE		88.27		0.0670		0.0660		0.0665		5.870
SOLUTION RECIRCULÉE		62.22		0.0130		0.0120		0.0125		0.778
SOLUTION PAUVRE		114.34		0.0002		0.0002		0.0002		0.023
CLARIFICATEUR		31.62		0.0670		0.0660		0.0665		2.103
FORAGE #1										
FORAGE #2										
RES. TROP PLEIN										

SOUS-TOTAL TONNES 2269.83 1959.82

SOUS-TOTAL ONCES 10.900 90.229

BILAN MÉTALURGIQUE DE L'OR AVANT TAURUS

EPAISSISSEUR NO.1

ECHANTILLONS HAUTEUR DE	A	TONNES SOLIDES	TONNES LIQUIDES	CAMFLO TENEUR SOLIDES	CAMFLO TENEUR LIQUIDES	TAURUS TENEUR SOLIDES	TAURUS TENEUR LIQUIDES	TENEUR MOYENNE SOLIDES	TENEUR MOYENNE LIQUIDES	ONCES SOLIDES	ONCES LIQUIDES
	33		94.80		0.0650		0.0640		0.0645		6.115
33	50		48.84		0.0655		0.0650		0.0653		3.189
50	66	0.36	45.84	0.0030	0.0655	0.0020	0.0653	0.0025	0.0654	0.001	2.998
66	99	2.95	93.75	0.0025	0.0655	0.0023	0.0648	0.0024	0.0651	0.007	6.103
99	131	23.95	83.38	0.0023	0.0660	0.0028	0.0653	0.0025	0.0656	0.060	5.470
	CONE	16.53	17.08	0.0100	0.0660	0.0103	0.0655	0.0101	0.0658	0.167	1.124
	))TAL TONNES	43.79	383.69					SOUS-TOTAL ONCES		0.235	24.999

EPAISSISSEUR NO.2

ECHANTILLONS HAUTEUR DE	A	TONNES SOLIDES	TONNES LIQUIDES	CAMFLO TENEUR SOLIDES	CAMFLO TENEUR LIQUIDES	TAURUS TENEUR SOLIDES	TAURUS TENEUR LIQUIDES	TENEUR MOYENNE SOLIDES	TENEUR MOYENNE LIQUIDES	ONCES SOLIDES	ONCES LIQUIDES
	CONE										
	))TAL TONNES							SOUS-TOTAL ONCES			

EPAISSISSEUR NO.3

ECHANTILLONS HAUTEUR DE	A	TONNES SOLIDES	TONNES LIQUIDES	CAMFLO TENEUR SOLIDES	CAMFLO TENEUR LIQUIDES	TAURUS TENEUR SOLIDES	TAURUS TENEUR LIQUIDES	TENEUR MOYENNE SOLIDES	TENEUR MOYENNE LIQUIDES	ONCES SOLIDES	ONCES LIQUIDES
	33		94.80		0.0850		0.0850		0.0850		8.058
33	66		94.80		0.0850		0.0850		0.0850		8.058
66	72		17.24		0.0850		0.0850		0.0850		1.465
72	99	1.81	76.92	0.0020	0.0805	0.0020	0.0813	0.0020	0.0809	0.004	6.223
99	131	25.38	82.87	0.0025	0.0805	0.0028	0.0813	0.0026	0.0809	0.066	6.704
	CONE	15.82	17.33	0.0120	0.0850	0.0118	0.0850	0.0119	0.0850	0.188	1.473
	))TAL TONNES	43.01	383.96					SOUS-TOTAL ONCES		0.258	31.981

TONNES

SOLIDES	LIQUIDES	TOTAL
2356.63	2727.47	5084.10

ONCES

SOLIDES	LIQUIDES	TOTAL
11.393	147.209	158.602

TAURUS
BILAN APRÈS
16 SEPTEMBRE 2004

BILAN MÉTALURGIQUE DE L'OR

ECHANTILLONS	TONNES SOLIDES	TONNES LIQUIDES	CAMFLO TENEUR SOLIDES	CAMFLO TENEUR LIQUIDES	TAURUS TENEUR SOLIDES	TAURUS TENEUR LIQUIDES	TENEUR MOYENNE SOLIDES	TENEUR MOYENNE LIQUIDES	ONCES SOLIDES	ONCES LIQUIDES
BROYEUR # 1	5.56	3.72	0.4455	0.0900	0.4660	0.0920	0.4558	0.0910	2.534	0.339
BROYEUR # 2	6.63	3.68	0.4260	0.0780	0.5435	0.0800	0.4848	0.0790	3.214	0.291
AGITATEUR # 1	371.12	305.18	0.0195	0.0810	0.0170	0.0800	0.0183	0.0805	6.791	24.667
AGITATEUR # 2	369.43	302.97	0.0120	0.1120	0.0125	0.1120	0.0123	0.1120	4.544	33.933
AGITATEUR # 3	331.27	279.80	0.0075	0.1250	0.0080	0.1270	0.0078	0.1260	2.584	35.255
AGITATEUR # 4	430.60	281.29	0.0065	0.0400	0.0060	0.0410	0.0063	0.0405	2.713	11.392
AGITATEUR # 5	372.75	248.36	0.0060	0.0530	0.0060	0.0530	0.0060	0.0530	2.237	13.163
AGITATEUR # 6	392.53	241.22	0.0060	0.0170	0.0050	0.0170	0.0055	0.0170	2.159	4.101
SOLUTION RICHE		88.27		0.0570		0.0570		0.0570		5.031
SOLUTION RECIRCULÉE		72.81		0.0170		0.0170		0.0170		1.238
SOLUTION PAUVRE		115.01		0.0002		0.0002		0.0002		0.023
CLARIFICATEUR		31.62		0.0570		0.0570		0.0570		1.802
PUISARD # 1										
PUISARD # 2										
RES. TROP PLEIN										

SOUS-TOTAL TONNES 2279.89 1973.93

SOUS-TOTAL ONCES 26.776 131.135

BILAN MÉTALURGIQUE DE L'OR APRÈS TAURUS

EPAISSISSEUR NO.1

ECHANTILLONS		TONNES		CAMFLO TENEUR		TAURUS TENEUR		TENEUR MOYENNE		ONCES	
DE	A	SOLIDES	LIQUIDES	SOLIDES	LIQUIDES	SOLIDES	LIQUIDES	SOLIDES	LIQUIDES	SOLIDES	LIQUIDES
	33		94.80		0.0660		0.0650		0.0655		6.209
33	66		94.80		0.0645		0.0640		0.0643		6.096
66	99	4.08	93.33	0.0060	0.0635	0.0050	0.0630	0.0055	0.0633	0.022	5.908
99	131	43.52	76.22	0.0135	0.0655	0.0135	0.0645	0.0135	0.0650	0.588	4.954
	CONE	20.32	15.65	0.0225	0.0670	0.0235	0.0660	0.0230	0.0665	0.467	1.041
	)TAL TONNES	67.92	374.80					SOUS-TOTAL ONCES		1.077	24.208

EPAISSISSEUR NO.2

ECHANTILLONS HAUTEUR		TONNES		CAMFLO TENEUR		TAURUS TENEUR		TENEUR MOYENNE		ONCES	
DE	A	SOLIDES	LIQUIDES	SOLIDES	LIQUIDES	SOLIDES	LIQUIDES	SOLIDES	LIQUIDES	SOLIDES	LIQUIDES
	CONE										
	)TAL TONNES							SOUS-TOTAL ONCES			

EPAISSISSEUR NO.3

ECHANTILLONS HAUTEUR		TONNES		CAMFLO TENEUR		TAURUS TENEUR		TENEUR MOYENNE		ONCES	
DE	A	SOLIDES	LIQUIDES	SOLIDES	LIQUIDES	SOLIDES	LIQUIDES	SOLIDES	LIQUIDES	SOLIDES	LIQUIDES
					0.0525		0.0525		0.0525		
	33		94.8		0.0525		0.0525		0.0525		4.977
33	66		94.8		0.0495		0.0490		0.0493		4.674
66	99	28.93	84.36	0.0220	0.0475	0.0215	0.0470	0.0218	0.0473	0.631	3.990
99	131	60.78	69.99	0.0235	0.0490	0.0228	0.0490	0.0231	0.0490	1.404	3.430
	CONE	17.98	16.49	0.0253	0.0500	0.0265	0.0500	0.0259	0.0500	0.466	0.825
	)TAL TONNES	107.69	360.44					SOUS-TOTAL ONCES		2.501	17.896

TONNES

SOLIDES	LIQUIDES	TOTAL
2455.50	2709.17	5164.67

ONCES

SOLIDES	LIQUIDES	TOTAL
30.354	173.239	203.593

TAURUS

INVENTAIRE AVANT

06 SEPTEMBRE 2004

INVENTAIRE AGITATEUR

	HAUTEUR DE RÉFÉRENCE	HAUTEUR DE LA DIFFÉRENCE	HAUTEUR DE PULPE	PIEDS CUBE AU PIED	PIEDS CUBES TOTAL	DENSITÉ KG/LITRE	TONNES TOTALES	% SOLIDE	TONNES SOLIDES	TONNES LIQUIDES
1	25.11458	2.29167	22.82291	617.5877	14095.15	1.56000	686.33	55.84%	383.25	303.08
2	26.30208	3.66667	22.63541	617.5877	13979.35	1.57000	685.06	56.48%	386.89	298.17
3	26.34375	5.93750	20.40625	617.5877	12602.65	1.56500	615.62	56.16%	345.73	269.89
4	26.36458	3.70833	22.65625	617.5877	13992.22	1.62000	707.52	59.53%	421.21	286.31
5	26.38542	6.58333	19.80209	617.5877	12229.53	1.55500	593.58	55.52%	329.55	264.03
6	26.32292	6.91667	19.40625	617.5877	11985.06	1.67000	624.74	62.41%	389.89	234.85
SURPLUS										
					TOTAL ⟶		3912.85		2256.52	1656.33

RÉSERVOIR DE SOLUTION

	HAUTEUR DE RÉFÉRENCE	HAUTEUR DE LA DIFFÉRENCE	HAUTEUR LIQUIDE	PIEDS CUBE AU PIED	PIEDS CUBES TOTAL	TONNES LIQUIDES
JTION RICHE	15.37500	1.31000	14.06500	201.0600	2827.91	88.27
RIFICATEUR					1013.04	31.62
ION PAUVRE	15.43750	1.04167	14.39583	254.4696	3663.30	114.34
SOLUTION RECIRCULÉE	15.37500	7.54167	7.83333	254.4696	1993.34	62.22
TROP PLEIN						
					TOTAL ⟶	296.45

BROYEUR

	PIEDS CUBES TOTAL	DENSITÉ KG / LITRE	TONNES TOTALES	% SOLIDE	TONNES SOLIDES	TONNES LIQUIDES
BROYEUR #1	183.37	1.70000	9.73	64.05%	6.23	3.50
BROYEUR #2	194.51	1.75000	10.62	66.67%	7.08	3.54
	TOTAL ⟶		20.350		13.31	7.04

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ÉPAISSISSEUR NO. 1

DE	A	HAUTEUR CALCULÉE	VOLUME AU PIED	PIEDS CUBE	DENSITÉ	TONNES TOTALES	% SOLIDE	TONNES SOLIDE	TONNES LIQUIDES
	2.75000	2.75000	1104.47	3037.29	1.0000	94.80			94.80
2.75000	4.16667	1.41667	1104.47	1564.67	1.0000	48.84			48.84
4.16667	5.50000	1.33333	1104.47	1472.62	1.0050	46.20	0.77%	0.36	45.84
5.50000	8.25000	2.75000	1104.47	3037.29	1.0200	96.70	3.05%	2.95	93.75
8.25000	10.91667	2.66667	1104.47	2945.26	1.1675	107.33	22.32%	23.95	83.36
	CONE			736.31	1.4625	33.61	49.19%	16.53	17.08
BILAN TOTAL ÉPAISSISSEUR NO. 1				12793.44		427.48		43.79	383.69

ÉPAISSISSEUR NO. 2

DE	A	HAUTEUR CALCULÉE	VOLUME AU PIED	PIEDS CUBE	DENSITÉ	TONNES TOTALES	% SOLIDE	TONNES SOLIDE	TONNES LIQUIDES
	CONE				1.0000				
BILAN TOTAL ÉPAISSISSEUR NO. 2									

ÉPAISSISSEUR NO. 3

DE	A	HAUTEUR CALCULÉE	VOLUME AU PIED	PIEDS CUBE	DENSITÉ	TONNES TOTALES	% SOLIDE	TONNES SOLIDE	TONNES LIQUIDES
	2.75000	2.75000	1104.47	3037.29	1.0000	94.80			94.80
2.75000	5.50000	2.75000	1104.47	3037.29	1.0000	94.80			94.80
5.50000	6.00000	0.50000	1104.47	552.24	1.0000	17.24			17.24
6.00000	8.25000	2.25000	1104.47	2485.06	1.0150	78.73	2.30%	1.81	76.92
8.25000	10.91667	2.66667	1104.47	2945.26	1.1775	108.25	23.45%	25.38	82.87
	CONE			736.31	1.4425	33.15	47.72%	15.82	17.33
BILAN TOTAL ÉPAISSISSEUR NO. 3				12793.45		426.97		43.01	383.96

TOTAL TONNES SOLIDES 2356.63

TOTAL TONNES LIQUIDES 2727.47

TAURUS
INVENTAIRE APRÈS
16 SEPTEMBRE 2004

INVENTAIRE AGITATEUR

	HAUTEUR DE RÉFÉRENCE	HAUTEUR DE LA DIFFÉRENCE	HAUTEUR DE PULPE	PIEDS CUBE AU PIED	PIEDS CUBES TOTAL	DENSITÉ KG/LITRE	TONNES TOTALES	% SOLIDE	TONNES SOLIDES	TONNES LIQUIDES
1	25.11458	2.33333	22.78125	617.5877	14069.42	1.54000	676.30	54.88%	371.12	305.18
2	26.30208	3.66667	22.63541	617.5877	13979.35	1.54100	672.40	54.94%	369.43	302.97
3	26.34375	5.62500	20.71875	617.5877	12795.65	1.53000	611.07	54.21%	331.27	279.80
4	26.36458	3.70833	22.65625	617.5877	13992.22	1.63000	711.89	60.49%	430.60	281.29
5	26.38542	6.52083	19.86459	617.5877	12268.13	1.62200	621.11	60.01%	372.75	248.36
6	26.32292	6.45833	19.86459	617.5877	12268.13	1.65500	633.75	61.94%	392.53	241.22
SURPLUS										
TOTAL →							3926.52		2267.70	1658.82

RÉSERVOIR DE SOLUTION

	HAUTEUR DE RÉFÉRENCE	HAUTEUR DE LA DIFFÉRENCE	HAUTEUR LIQUIDE	PIEDS CUBE AU PIED	PIEDS CUBES TOTAL	TONNES LIQUIDES
...TION RICHE	15.37500	1.31000	14.06500	201.06000	2827.91	88.27
...RIFICATEUR					1013.04	31.62
...ION PAUVRE	15.43750	0.95833	14.47917	254.46960	3684.51	115.01
SOLUTION RECIRCULÉE	15.37500	6.20833	9.16667	254.46960	2332.64	72.81
TROP PLEIN						
TOTAL →						307.71

BROYEUR

	PIEDS CUBES TOTAL	DENSITÉ KG / LITRE	TONNES TOTALES	% SOLIDE	TONNES SOLIDES	TONNES LIQUIDES
BROYEUR #1	163.37	1.62100	9.28	59.95%	5.56	3.72
BROYEUR #2	194.51	1.69800	10.31	64.33%	6.63	3.68
TOTAL →			19.59		12.19	7.40

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ÉPAISSISSEUR NO. 1

DE	A	HAUTEUR CALCULÉE	VOLUME AU PIED	PIEDS CUBE	DENSITÉ	TONNES TOTALES	% SOLIDE	TONNES SOLIDE	TONNES LIQUIDES
	2.75000	2.75000	1104.47	3037.29	1.0000	94.80			94.80
2.75000	5.50000	2.75000	1104.47	3037.29	1.0000	94.80			94.80
5.50000	8.25000	2.75000	1104.47	3037.29	1.0275	97.41	4.19%	4.08	93.33
8.25000	10.91667	2.66667	1104.47	2945.26	1.3025	119.74	36.35%	43.52	76.22
	CONE			736.31	1.5650	35.97	56.50%	20.32	15.65
BILAN TOTAL EPAISSISSEUR NO. 1				12793.44		35.97		67.92	374.80

ÉPAISSISSEUR NO. 2

DE	A	HAUTEUR CALCULÉE	VOLUME AU PIED	PIEDS CUBE	DENSITÉ	TONNES TOTALES	% SOLIDE	TONNES SOLIDE	TONNES LIQUIDES
	CONE				1.0000				
BILAN TOTAL EPAISSISSEUR NO. 2									

ÉPAISSISSEUR NO. 3

DE	A	HAUTEUR CALCULÉE	VOLUME AU PIED	PIEDS CUBE	DENSITÉ	TONNES TOTALES	% SOLIDE	TONNES SOLIDE	TONNES LIQUIDES
	2.75000	2.75000	1104.47	3037.29	1.0000	94.80			94.80
2.75000	5.50000	2.75000	1104.47	3037.29	1.0000	94.80			94.80
5.50000	8.25000	2.75000	1104.47	3037.29	1.1950	113.29	25.54%	28.93	84.36
8.25000	10.91667	2.66667	1104.47	2945.26	1.4225	130.77	46.48%	60.78	69.99
	CONE			736.31	1.5000	34.47	52.17%	17.98	16.49
BILAN TOTAL EPAISSISSEUR NO. 3				12793.44		468.13		107.69	360.44

TOTAL TONNES SOLIDES 2455.30

TOTAL TONNES LIQUIDES 2709.17

INVENTAIRE APRES

Scellé valve fermée (camion):

Scellé valve ouverte(client) :

Heure lecture de la balance : 10:45 Lecture : 1006228.6

Lecture du débimètre : 468871

Heure valve client ouverte : 10:47

NOM DE LA MINE: **TAURUS**

DATE: 16 SEPTEMBRE 2004

Après		Mesure	Densité
S.G:	2.77	En	En
		Pouces	Kilo/Litre
Agitateur No 1		28.00	1.5400
2		44.00	1.5410
3		67.50	1.5300
4		44.50	1.6300
5		76.25	1.6220
6		77.50	1.6550
Solution riche		15.72	1.0000
Clarificateur			1.0000
Solution pauvre		11.50	1.0000
Solution recirculée		74.50	1.0000
Réservoir de trop plein			
Broyeur No 1			1.6210
Broyeur No 2			1.6980
Épaississeurs No 1 a			1.0000
b			1.0000
c			1.0000
d		33.00	1.0000
e		66.00	1.0000
f		99.00	1.0550
g		131.00	1.5500
Cone			1.5800
Épaississeurs No 2 a			1.0000
b			1.0000
c			1.0000
d			1.0000
e			1.0000
f			1.0000
g			1.0000
Cone			1.0000
Épaississeurs No 3 a			1.0000
b			1.0000
c			1.0000
d		33.00	1.0000
e		66.00	1.0000
f		99.00	1.3900
g		131.00	1.4550
Cone			1.5450

ANALYSE INVENTAIRE AVANT SELON: CAMFLO

PULPE:

		ANALYSE		[illegible]
AGITATEUR	#1 →	[illegible]	[illegible]	[illegible]
	#2	0.0030	0.0020	[illegible]
	#3	0.0020	0.0020	[illegible]
	#4	0.0020	0.0020	[illegible]
	#5	0.0020	0.0020	[illegible]
	#6	0.0020	0.0020	[illegible]
ÉPAISSISSEUR SOUVERSE	#1 →	0.0170	0.0180	[illegible]
FOND	#1	0.0020	0.0030	[illegible]
NIVEAU	90	0.0020	0.0020	[illegible]
NIVEAU	60	0.0030	0.0030	[illegible]
NIVEAU				
ÉPAISSISSEUR SOUVERSE	#2 →			
FOND	#2			
NIVEAU	90			
NIVEAU	33			
NIVEAU				
ÉPAISSISSEUR SOUVERSE	#3 →	0.0220	0.0200	[illegible]
FOND	#3	0.0030	0.0030	[illegible]
NIVEAU	90	0.0020	0.0020	[illegible]
NIVEAU	33			
NIVEAU				
BROYEUR	#1 →	0.5000	0.5950	[illegible]
	#2	0.3000	0.2960	[illegible]

SOLUTION:

		ANALYSE		[illegible]
ÉPAISSISSEUR SURVERSE	#1 →	0.0650	0.0650	[illegible]
NIVEAU	33	0.0650	0.0650	[illegible]
NIVEAU	90	0.0660	0.0660	[illegible]
NIVEAU	66	0.0650	0.0650	[illegible]
NIVEAU	90	0.0660	0.0660	[illegible]
FOND	CÔNE)	0.0660	0.0660	[illegible]
ÉPAISSISSEUR SURVERSE	#2 →			
NIVEAU				
NIVEAU	33			
INTERFACE	66			
NIVEAU	90			
FOND	CÔNE)			
ÉPAISSISSEUR SURVERSE	#3 →	0.0850	0.0850	[illegible]
NIVEAU	33	0.0850	0.0850	[illegible]
NIVEAU	66	0.0850	0.0850	[illegible]
NIVEAU	72	0.0850	0.0850	
NIVEAU	90	0.0760	0.0760	[illegible]
FOND	CÔNE)	0.0850	0.0850	[illegible]
BROYEUR	#1 →	0.3400	0.3400	[illegible]
	#2	0.1900	0.1900	[illegible]
AGITATEUR	#1 →	0.0780	0.0780	[illegible]
	#2	0.0040	0.0040	[illegible]
	#3	0.0850	0.0850	[illegible]
	#4	0.0140	0.0140	[illegible]
	#5	0.0130	0.0130	[illegible]
	#6	0.0040	0.0040	[illegible]
SOLUTION	RICHE	0.0670	0.0670	[illegible]
SOLUTION	RECIRC.	0.0130	0.0130	[illegible]
SOLUTION	PAUVRE	0.0002	0.0002	[illegible]

ANALYSE INVENTAIRE AVANT SELON : TAURUS

PULPE:

		ANALYSE		[illegible]
AGITATEUR	#1 →	[illegible]	0.0040	[illegible]
	#2	0.0020	0.0020	[illegible]
	#3	0.0020	0.0020	[illegible]
	#4	0.0020	0.0020	[illegible]
	#5	0.0020	0.0020	[illegible]
	#6	0.0020	0.0020	[illegible]
ÉPAISSISSEUR SOUVERSE	#1 →	0.0170	0.0180	[illegible]
FOND	#1	0.0030	0.0030	[illegible]
NIVEAU	90	0.0030	0.0020	[illegible]
NIVEAU	68	0.0020	0.0020	[illegible]
NIVEAU				
ÉPAISSISSEUR SOUVERSE	#2 →			
FOND	#2			
NIVEAU	90			
NIVEAU	60			
NIVEAU				
ÉPAISSISSEUR SOUVERSE	#3 →	0.0200	0.0200	[illegible]
FOND	#3	0.0030	0.0040	[illegible]
NIVEAU	90	0.0020	0.0020	[illegible]
NIVEAU	66			
NIVEAU				
BROYEUR	#1 →	0.5720	0.5950	[illegible]
	#2	0.3270	0.3050	[illegible]

SOLUTION:

		ANALYSE		[illegible]
ÉPAISSISSEUR SURVERSE	#1 →	0.0640	0.0640	[illegible]
NIVEAU	33	0.0640	0.0640	[illegible]
NIVEAU	90	0.0660	0.0660	[illegible]
NIVEAU	66	0.0640	0.0650	
NIVEAU	90	0.0630	0.0650	[illegible]
FOND	CÔNE)	0.0659	0.0660	[illegible]
ÉPAISSISSEUR SURVERSE	#2 →			
NIVEAU				
NIVEAU	33			
INTERFACE	66			
NIVEAU	90			
FOND	CÔNE)			
ÉPAISSISSEUR SURVERSE	#3 →	0.0850	0.0850	[illegible]
NIVEAU	33	0.0850	0.0850	[illegible]
NIVEAU	66	0.0850	0.0850	[illegible]
NIVEAU	72	0.0850	0.0859	
NIVEAU	90	0.0770	0.0780	[illegible]
FOND	CÔNE)	0.0850	0.0856	[illegible]
BROYEUR	#1 →	0.3410	0.3430	[illegible]
	#2	0.1980	0.1980	[illegible]
AGITATEUR	#1 →	0.0770	0.0770	0.0770
	#2	0.0830	0.0830	[illegible]
	#3	0.0840	0.0840	[illegible]
	#4	0.0130	0.0130	[illegible]
	#5	0.0140	0.0140	[illegible]
	#6	0.0040	0.0040	[illegible]
SOLUTION	RICHE	0.0660	0.0660	0.0660
SOLUTION	RECIRC.	0.0120	0.0120	0.0120
SOLUTION	PAUVRE	0.0002	0.0002	0.0002

ANALYSE INVENTAIRE APRÈS SELON : CAMFLO

PULPE:

		ANALYSE		[illegible]
AGITATEUR	#1	0.0130	[illegible]	[illegible]
	#2	0.0120	0.0120	[illegible]
	#3	0.0070	[illegible]	[illegible]
	#4	0.0070	0.0060	[illegible]
	#5	0.0060	0.0060	[illegible]
	#6	0.0060	[illegible]	[illegible]
ÉPAISSISSEUR SOUVERSE	#1	0.0240	0.0240	0.0225
FOND	#1	0.0210	0.0210	0.0135
NIVEAU	99	0.0060	0.0060	[illegible]
NIVEAU	66			
NIVEAU	33			
ÉPAISSISSEUR SOUVERSE	#2			
FOND	#2			
NIVEAU	99			
NIVEAU	66			
NIVEAU				
ÉPAISSISSEUR SOUVERSE	#3	0.0260	0.0250	[illegible]
FOND	#3	0.0250	0.0250	[illegible]
NIVEAU	99	0.0230	0.0210	[illegible]
NIVEAU	66			
NIVEAU				
BROYEUR	#1	0.4570	0.4340	[illegible]
	#2	0.4100	0.4420	[illegible]

SOLUTION:

		ANALYSE		[illegible]
ÉPAISSISSEUR SURVERSE	#1	0.0660	0.0660	[illegible]
NIVEAU				[illegible]
NIVEAU	33	0.0650	0.0640	
NIVEAU	66	0.0630	0.0630	[illegible]
NIVEAU	99	0.0640	0.0640	[illegible]
FOND (CÔNE)		0.0670	0.0670	0.0670
ÉPAISSISSEUR SURVERSE	#2			
NIVEAU				
NIVEAU	33			
NIVEAU	66			
NIVEAU	99			
FOND (CÔNE)				
ÉPAISSISSEUR SURVERSE	#3	0.0530	0.0530	[illegible]
NIVEAU				[illegible]
NIVEAU	33	0.0520	0.0520	[illegible]
NIVEAU	66	0.0470	0.0470	0.0475
NIVEAU	99	0.0480	0.0480	[illegible]
FOND (CÔNE)		0.0500	0.0500	[illegible]
BROYEUR	#1	0.0900	0.0900	[illegible]
	#2	0.0780	0.0780	0.0780
AGITATEUR	#1	0.0810	0.0810	[illegible]
	#2	0.1120	0.1120	0.1120
	#3	0.1250	0.1250	0.1250
	#4	0.0400	0.0400	[illegible]
	#5	0.0530	0.0530	0.0530
	#6	0.0170	0.0170	0.0170
SOLUTION	RICHE	0.0570	0.0570	0.0570
SOLUTION	RECIRC.	0.0170	0.0170	0.0170
SOLUTION	PAUVRE	0.0002	0.0002	0.0002

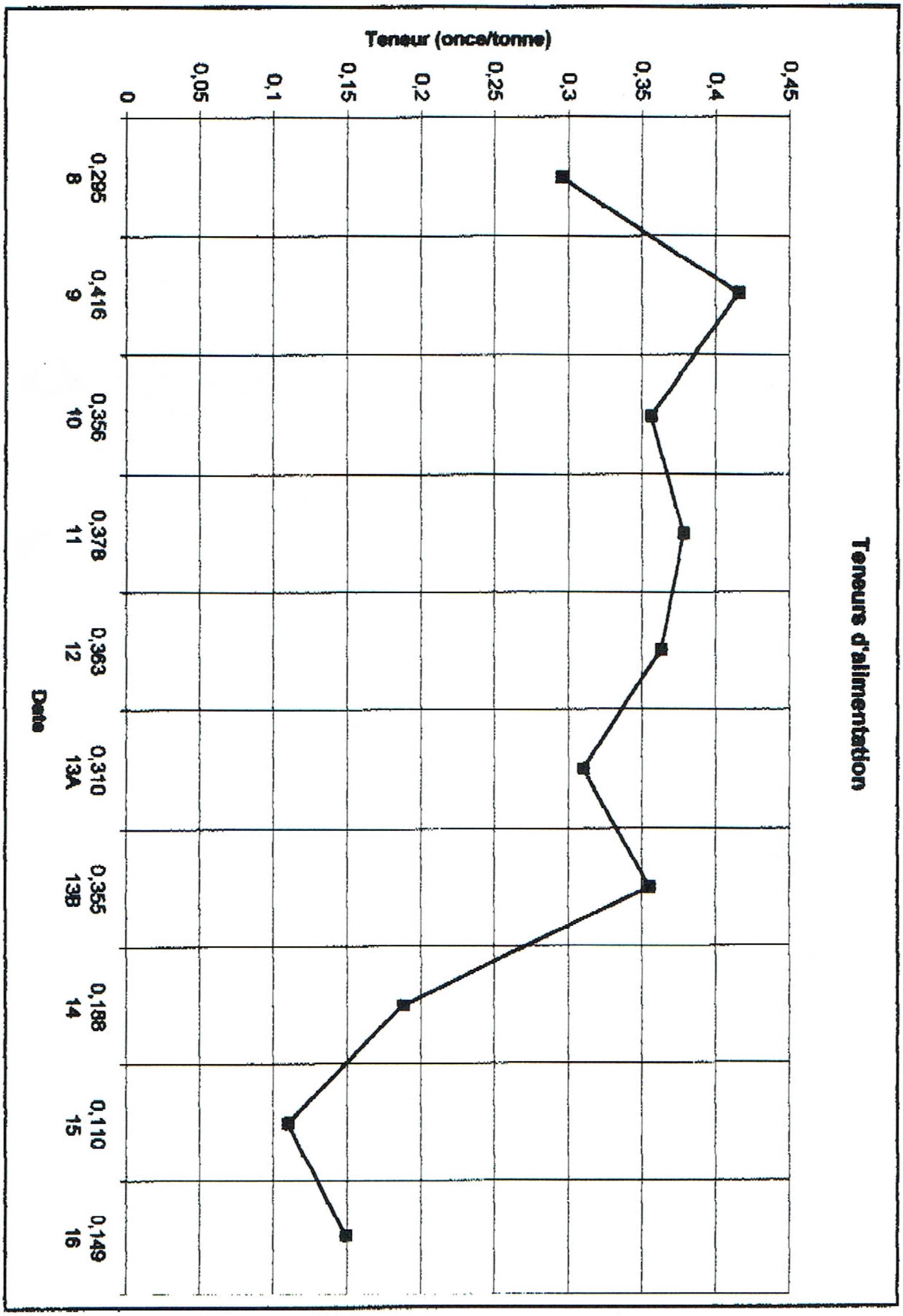
ANALYSE INVENTAIRE APRÈS SELON : TAURUS

PULPE:

		ANALYSE		[illegible]
AGITATEUR	#1	[illegible]	0.0160	[illegible]
	#2	0.0120	0.0130	[illegible]
	#3	0.0080	0.0080	[illegible]
	#4	0.0060	0.0060	[illegible]
	#5	0.0060	0.0060	[illegible]
	#6	0.0050	0.0050	[illegible]
ÉPAISSISSEUR SOUVERSE	#1	0.0250	0.0250	[illegible]
FOND	#1	0.0220	0.0220	0.0135
NIVEAU	99	0.0050	0.0050	[illegible]
NIVEAU	66			
NIVEAU	33			
ÉPAISSISSEUR SOUVERSE	#2			
FOND	#2			
NIVEAU	99			
NIVEAU	66			
NIVEAU				
ÉPAISSISSEUR SOUVERSE	#3	0.0290	0.0290	[illegible]
FOND	#3	0.0230	0.0250	[illegible]
NIVEAU	99	0.0210	0.0220	[illegible]
NIVEAU	66			
NIVEAU				
BROYEUR	#1	0.4180	0.5140	[illegible]
	#2	0.5570	0.5300	0.5435

SOLUTION:

		ANALYSE		[illegible]
ÉPAISSISSEUR SURVERSE	#1	0.0630	0.0630	[illegible]
NIVEAU				0.0660
NIVEAU	33	0.0650	0.0635	0.0640
NIVEAU	66	0.0630	0.0630	[illegible]
NIVEAU	99	0.0620	0.0630	0.0645
FOND (CÔNE)		0.0660	0.0660	[illegible]
ÉPAISSISSEUR SURVERSE	#2			
NIVEAU				
NIVEAU	33			
NIVEAU	66			
NIVEAU	99			
FOND (CÔNE)				
ÉPAISSISSEUR SURVERSE	#3	0.0530	0.0530	[illegible]
NIVEAU				[illegible]
NIVEAU	33	0.0520	0.0520	[illegible]
NIVEAU	66	0.0460	0.0460	[illegible]
NIVEAU	99	0.0480	0.0480	[illegible]
FOND (CÔNE)		0.0500	0.0500	[illegible]
BROYEUR	#1	0.0920	0.0920	[illegible]
	#2	0.0800	0.0800	[illegible]
AGITATEUR	#1	0.0800	0.0800	[illegible]
	#2	0.1120	0.1120	0.1120
	#3	0.1270	0.1270	0.1270
	#4	0.0410	0.0410	[illegible]
	#5	0.0530	0.0530	[illegible]
	#6	0.0170	0.0170	0.0170
SOLUTION	RICHE	0.0570	0.0570	[illegible]
SOLUTION	RECIRC.	0.0170	0.0170	0.0170
SOLUTION	PAUVRE	0.0002	0.0002	[illegible]


Teneurs d'alimentation
Teneur (once/tonne)
0
0,05
0,1
0,15
0,2
0,25
0,3
0,35
0,4
0,45
0,295
8
0,416
9
0,356
10
0,378
11
0,363
12
0,310
13A
0,355
13B
0,188
14
0,110
15
0,149
16
Date

INTRODUCTION

Nous avons été mandatés par M. André Deguise ing, afin de vérifier le bon déroulement de l'usinage d'un lot de minerai provenant de la propriété Fenelon. Ceci a impliqué une première visite pour discuter de la méthode d'échantillonnage, d'un inventaire de départ de l'usinage du lot, de visite journalière aléatoire, la prise d'échantillons et finalement à assister à l'ouverture des presses, à la coulée de la brique d'or et au nettoyage de la raffinerie.

Le but de ce rapport est de permettre à la prochaine personne qui suivra un lot de minerai provenant de Fenelon d'avoir en main tous les éléments concernant l'usinage à forfait.

1. Visite initiale.

On a établit que les étapes de l'inventaire sont les suivantes :

- Prendre les mesures de la quantité de minerai qui reste dans le silo à minerai grossier. Une fois cette mesure prise, on peu commencé à entrer le minerai dans le silo à minerai grossier.
- On vérifie la propreté du circuit de concassage.
- Vérifier le zéro et le totalisateur sur la balance du convoyeur qui amène le minerai dans le silo à minerai fin # 3.
- Vérifier si le silo à minerai fin est vide, si oui on peu commencer à le remplir.
- On mesure ensuite le niveau de pulpe dans les épaississeurs en prenant un échantillon à divers niveau dans la pulpe et à l'interface. On prend une mesure de densité pour chaque échantillon. On mesure le niveau d'eau dans l'épaississeur et on prend un échantillon de liquide en surface à divers point autour de l'épaississeur. On prend un échantillon de pulpe de la sousverse des épaississeurs à l'entrée de l'agitateur No 1. On analysera seulement le solide car la teneur du liquide serait dilué par l'eau de le gland de pompe.

- On mesure le niveau de pulpe dans chaque agitateur après avoir arrêté l'arrivé d'air dans l'agitateur. On prend un échantillon de pulpe qui sera analysé après une filtration pour le solide et le liquide. On prend aussi une mesure de la densité de la pulpe.
- On vérifie le niveau de pulpe dans les filtreurs à tambour qui normalement sont vide.
- On vérifie le niveau d'eau dans les puisards qui normalement devrait être vide.
- On vérifie le niveau d'eau dans les réservoirs de solution stérile et de solution riche et recirculé et on prend des échantillons.
- On échantillonne les broyeurs à boulet.
- Vérifier le totalisateur de la balance qui alimente le broyeur à barres.
- On démarre le circuit avec le nouveau minerai.
- On fait le changement de presse après la vérification des numéros des celées.

2. Inventaire (description)

2.1 Circuit de concassage

La visite a montré que le dessous des convoyeurs est très propre ainsi que l'entourage des concasseurs coniques et des deux tamis. Il y a un peu de minerai sur le convoyeur de recirculation des tamis.

Le silo de minerai grossier est aussi vide qu'il est possible de l'être. On prend des mesures permettant de vérifier le niveau de la charge morte au fond du silo.

On vérifie le zéro sur la balance qui amène le minerai concassé au silo de minerai fin. Il est important que la courroie du convoyeur soit mise en marche environ 2 heures avant la vérification du zéro. La marge d'erreur de la balance est de ½ de 1 %, ce qui signifie que la valeur que l'on doit obtenir lorsqu'on place le poids étalon se situe entre 13,534 et 13,670.

On commence par noter le chiffre du totalisateur. Dans notre cas il s'agit de 340 606,26 tonnes courtes. On descend le poids en place sur la balance, on démarre l'auto zéro et après 5 tours complet la balance affiche 13,435 ce qui est en dehors des limites. On enlève le poids sur la balance et on démarre l'auto zéro. La première valeur nous donne 0,219. On redémarre l'auto zéro et on obtient 0,002, on refait une tentative et on obtient 0,002, on refait une autre tentative et on obtient 0,003. On refait une dernière tentative qui donne 0,001 ce qui est acceptable.

On descend à nouveau le poids sur la balance, on démarre l'auto zéro et après 5 tours on obtient 13,653 ce qui est acceptable. On refait la mesure et on obtient 13,656. On vérifie le totalisateur à la fin de la vérification du zéro et on obtient 340 606,23 tonnes courtes.

On vérifie ensuite les silos de minerai fin. Le silo No 1 est plein à 100 % de minerai de Beaufor. Le silo No 2 est vide et les chutes sont ouvertes en permanence afin de permettre aux particules qui pourrait tomber du convoyeur qui passe au-dessus du silo de se mélanger au minerai qui est soutiré du silo No 3. Le silo No 3 est complètement vide. C'est le seul silo qui devrait recevoir le minerai de Fenelon.

2.2 Les réservoirs.

On passe ensuite aux épaississeur qui sont au nombre de trois. Actuellement on utilise que les épaississeurs 1 et 3. On échantillonne les épaississeurs de la façon suivante :

- À l'aide d'une tige de bois graduée sur laquelle est maintenu avec du ruban adhésif d'électricien un tuyau flexible qui servira à prélever des échantillons de pulpe à divers niveau dans les épaississeurs.
- On utilise deux équipes une sur l'épaississeur et l 'autre au pied de l'épaississeur. L'équipe du haut amorce le siphon et plonge la tige au fond de l'épaississeur, ce qui correspond à une profondeur prédéterminée. Il avise l'équipe du bas qu'il est près à maintenir la tige à cette position.
- L'équipe du bas laisse coulé la pulpe un moment puis prend une mesure de la densité et ensuite prend un échantillon. Elle avise ensuite l'équipe du haut qu'il peu changer de position.
- On recommence la séquence en remontant la tige à des niveaux pré établit jusqu'à ce qu'on arrive dans la zone d'eau claire. On redescend alors jusqu'à ce qu'on atteigne à nouveau la pulpe. On note alors le niveau de l'interface.
- On prend aussi un échantillon de la surverse à divers points autour de l'épaississeur.
- On échantillonne la sousverse de l'épaississeur mais seul le solide sera analysé car le liquide est dilué par le gland de pompe.

On échantillonne ensuite les agitateurs qui sont au nombre de 6. On arrête l'air de façon à mesurer le niveau réel de pulpe. On mesure ensuite la densité de la pulpe et on prend un échantillon. On répète la séquence pour les 6 agitateurs.

On vérifie ensuite le niveau de pulpe dans les bases de filtreurs. Dans les cas présent toutes les bases sont vides.

On vérifie ensuite le niveau des puisards qui sont tous vides dans le cas présent.

On vérifie le niveau dans les réservoirs de solution riche, pauvre et recirculé. On prend un échantillon dans chacun.

2.3 Les broyeurs

On échantillonne ensuite les broyeurs qui sont au nombre de 3. Puis on vérifie le nombre affiché sur le totalisateur de la balance qui alimente le broyeur à barre et on démarre le circuit avec le minerai de Fenelon.

2.4 Les presses

On fait le changement de presses après la vérification des numéros des celés. On applique de nouveaux celés sur les valves qui permette le changement de presse.

3. Opération

3.1 Suivit de l'opération

Une heure avant d'entrée le minerai de Fenelon dans le circuit, on a fait monter le niveau de cyanure à 1,2 livres par tonnes courtes. On a établit le tonnage de départ à 50 tonnes courtes par heure. On a aussi fait ajouter de l'oxygène dans l'agitateur No. 2. La première demi-journée (jeudi le 9 septembre) s'est passé sans problème.

La journée suivante (vendredi) j'ai fait une visite en début d'après midi. La pression avait monté anormalement dans les presses j'ai demandé à M. Gérald Lavoie d'arrêter d'ajouté de l'anti-précipitant dans le clarificateur. Il m'a dit qu'il le ferait.

Le lendemain (samedi) après midi, j'ai constaté que non seulement on n'avait pas arrêté d'ajouter l'anti précipitant mais en plus on avait ajouté du déincrustant, la pression avait continué, dans la soirée on avait secoué les presses en envoyant de l'air sous pression à l'intérieur. Le test de couleur a montré une perte d'or pendant 6 heures durant la nuit puis ça s'était replacé. J'ai demandé à l'opérateur des solutions d'arrêter d'ajouter tous ces produits. Il l'a fait tout de suite.

Le lendemain (dimanche) matin la pression avait continuer à monter dans les presses mais plus lentement. Le test de couleur n'avait pas montré de perte en or.

Le lendemain (lundi) matin le rapport d'opération a montré que plusieurs tests de couleur avaient révélé des pertes en or dans la soirée de dimanche. La pression continuait de monter dans les presses. Dans l'après midi, M. Richard Nolet m'a téléphoné pour me dire que les résultats d'analyse montraient d'énorme perte en or. On a décidé de changer de presse et de les vider le lendemain matin. M. Richard Nolet a fait baisser le tonnage pour minimiser les pertes en or.

Le lendemain (mardi) matin on a procédé à l'ouverture des presses. J'ai alors émis l'hypothèse que la monté de la pression des presses pourrait provenir de la formation d'hydroxyde de zinc, les hydroxydes étant très difficiles à filtrer. Pour vérifier cette hypothèse j'ai suggéré d'augmenter la quantité de sel de plomb ajouté avec la poudre de zinc, ce qui a été fait.

Pour ce qui est de l'ouverture des presses, la procédure a été la suivante :

Les mesures de sécurités qui entourent le vidage des presses sont maximums. Au départ il y a deux gardiens de l'agence Mirado qui sont là durant toute la durée du vidage des presses.

Pour accéder aux presses il faut que chaque personne aie une clef spéciale. Était présent, les deux gardiens, Gérald Lavoie contremaître d'opération et Edmond St-Jean ing consultant pour Fenelon plus quatre opérateurs qui ont vidé les presses.

Les celées sont vérifier, leurs numéros sont notés par les agents de sécurité avant qu'il ne soit coupé. Une grande panne est placée sous chaque presse pour recevoir le précipité. On ouvre alors les deux presses et un opérateur se place de chaque coté de chaque presse pour faire tomber le précipité dans chaque presse. Le coton qui est en contact avec le précipité est roulé de façon à ce que le précipité qui y est collé soit à l'intérieur puis placé dans un contenant à l'intérieur duquel on a étendu un sac de plastique noir.

Lorsque le contenant est plein, on place dessus un couvercle sur lequel on place deux celés. Cette opération est continuée jusqu'à ce que les deux presses soit vide. On enlève alors les pannes de sous les presses puis on place le précipité dans des contenants munis de couvercle qui visse et dans lesquels on a étendu un sac de plastique noir. Les pannes sont ensuite nettoyées avec des chiffons qui sont ensuite placé dans les contenants qui renferme les cotons de presse.

Tous les contenants qui renferme le précipité et les cotons de presse sont amené à la raffinerie sous la surveillance des agents de sécurité ainsi que de M. Lavoie et M. St-Jean.

Les opérateurs et un des agents de sécurité retournent dans la cage des presses pour remonter les presses. Il place les pannes sous les presses afin de recueillir le peu de précipité qui pourrait encore tomber. À la fin il nettoie les pannes avec des chiffons et il les place dans un coton qu'il roule en boule et qu'il maintienne en place à l'aide de ruban adhésif.

Le lendemain (mercredi) après midi, on a constaté que la pression avait très peu monté mais on avait apparition d'une bonne quantité de graphite dans les épaississeurs et les agitateurs. On a fait prendre des échantillons de graphite dans les agitateurs. J'ai aussi fait monter le tonnage afin de minimiser les surfaces exposées du graphite. J'ai eu une discussion avec M. Richard Nolet et M. Gérald Lavoie au sujet de la teneur en or de la solution pauvre. On est venu à la conclusion que le peu de temps ou il est noter qu'il y a des pertes d'or sur les rapports des opérateurs aux solutions sont très loin de justifier les teneurs obtenues par analyse et qui sont 25 fois supérieures aux teneurs normales. Pour obtenir des teneurs semblables, il faudrait que les opérateurs aient noter des pertes d'or à tous leurs tests, ce qui est loin d'être le cas.

Le lendemain (jeudi) On à fait l'inventaire et on a mis fin à l'alimentation du circuit avec le minerai de Fenelon à 10 heure 45.

4. Inventaire (discussion)

4.1 Inventaire No. 1

Lors du premier inventaire, il ne s'est rien passé de notable, leur méthode d'échantillonnage est très efficace et très bien rôdé. On a vérifier s'il y avait des accumulations de minerai sous le tamis qui recueille les bouts de plastique provenant du dynamitage à la surverse du cyclone. Il n'y avait aucune accumulation notable. Par contre, la grille de calcul qui sert à faire le bilan des inventaires fait très peu professionnelle. On y retrouve des volumes exprimés en pieds cube par pieds. Ces mêmes pieds cubes par pied sont multipliés par des kilogrammes par litre pour finir par donner des tonnes courtes après avoir passé par une série de facteur de conversion.

À près vérification, l'erreur de ces facteurs de conversion donne une erreur maximum inférieure à 0,2 % ce qui est quand même très acceptable. De plus, comme cette erreur est la même pour l'inventaire avant et après elle s'annule donc.

4.2 Inventaire No. 2

Lors de l'inventaire No.2 il a fallut plusieurs heures pour vider le silo No 3 car les fines particules avaient formé un monticule qui ne voulait pas descendre avec la simple vibration. On a re- vérifié s'il y avait des accumulations de minerai sous le tamis qui recueille les bouts de plastique provenant du dynamitage à la surverse du cyclone. Il n'y avait aucune accumulation notable.

Tout c'est très bien passé lors de l'inventaire, il n'y a rien de notable a souligné à l'exception de la difficulté à vidé le silo No. 3.

5. Le raffinage

Le raffinage s'est déroulé sous la surveillance de deux agents de sécurité de la firme Mirado. On a d'abord vidé les presses pendant que deux opérateurs partaient les fournaises et brûlaient les cotons des presses du changement de presses précédent.

Une fois les presses vidées on est tous partie pour la raffinerie avec le précipité recueillit dans des barils de plastique. Les deux opérateurs ont procédé au mélange du précipité avec les fondants. Cette opération c'est effectué de façon très efficace.

Lors de la coulée de la première brique, on a eu la désagréable constatation de voir un morceau du ciment réfractaire se détacher de la coquille du four. Ceci implique qu'on a dû continuer le raffinage avec une seule fournaise.

À partir de la deuxième brique, on a vu apparaître de la matte. La quantité de matte est allée en augmentant pour la troisième et la quatrième brique. Ceci simplement parce qu'on ajoutait la scorie riche qui venait avec la brique dans le four qui donnait la brique suivante.

À par ces quelques événements, il ne s'est rien passé de notable lors du raffinage. Chaque brique a été marquée puis pesé lors de la coulée de la dernière brique on a récupéré un bouton de 921,9 grammes et lors de la coulé de nettoyage de la fournaise on a récupéré un bouton de 207,1 grammes. Le total des quatre briques représente un poids de 3427,6 onces troy.

6. Nettoyage de la raffinerie

Le nettoyage de la raffinerie a débuté lundi le 20 septembre à 8 heures AM sous la surveillance d'un agent de sécurité. On a d'abord pesé la scorie pauvre qu'on a placé dans deux barils bruns, numérotés 469 et 470 sur lesquels on a placé des celées.

Pour les scories riches, on a enlevé tous les morceaux de matte que l'on a placé dans une chaudière et auquel on a ajouté le nettoyage des moules des briques. Cette chaudière a été pesée puis celées et marqué au nom de Taurus. Le reste de la scorie riche a été placé dans des chaudières qui ont été pesées puis celées et marqué au nom de Taurus. Ces chaudières sont conservées dans la raffinerie.

Finalement, tout l'intérieur de la hotte des fournaises a été gratté et un jet d'air a été passé de façon à récupérer tous les petits morceaux de scorie. Le tout a été soigneusement récupéré et a été placé dans une chaudière qui a été pesée puis celée et marquée au nom de Taurus.

7. Teneurs d'alimentation

Ce lot de minerai est constitué par deux zones différentes soit une zone pauvre et une zone riche. Il est difficile de faire une séparation précise entre les deux zones pendant l'opération. Cependant, on peut voir une variation dans les teneurs d'alimentation (tableau 1) qui peut permettre une certaine évaluation.

Tableau 1

Date	Tonnage sec Tc	teneur d'alimentation once/tc	récupération %
8	548	0,295	98,51
9	1194	0,416	98,25
10	1120	0,356	97,67
11	1186	0,378	94,27
12	1192	0,363	91,49
13A	732	0,310	93,48
13B	382	0,355	94,20
14	1018	0,188	94,19
15	1111	0,110	96,61
16	522	0,149	97,31

À partir de ce tableau on peut calculer qu'on a traité environ 6354 tonnes courtes de minerai riche dont la teneur devrait être aux environs de 0,362 once par tonne courte. Le minerai pauvre représente environ 2651 tonnes courtes avec une teneur d'environ 0,148 once par tonne.

8. Indice de Bond

Il est pratique de profité de l'occasion pour déterminer un indice de Bond dynamique à partir de la granulométrie d'entrée et de sortie du circuit de broyage en tenant compte du tonnage horaire, du voltage des broyeurs et de l'ampérage de chaque broyeur. Au moment du test on avait les paramètres suivants :

- voltage	2300 volts
- ampérage du broyeur à barre	60 ampères
- ampérage du broyeur à boulet No 1	83 ampères
- ampérage du broyeur à boulet N0 2	84 ampères
- tonnage horaire	50 tc/h
- D80	10 000 microns
- d80	70 microns

Avec le voltage, l'ampérage et le tonnage horaire on peut calculer que le circuit consomme 10,44 kw-h / tc de la façon suivante :

2300 volt * (60+83+84)A = 522 100 watt

donc, 522,1 kw divisé par 50 tc/ h = 10,44 kw-h / tc

À partir de cette consommation et en tenant compte de la granulométrie d'entrée et de sortie du circuit de broyage on peut calculer l'indice de Bond (Wi) de la façon suivante :

$$\frac{10\ Wi}{(70)^{0,5}} - \frac{10\ Wi}{(10\ 000)^{0,5}} = 10,44\ \text{kw-h / tc}$$

$$\frac{10\ Wi}{8,37} - \frac{10\ Wi}{100} = 10,44\ \text{kw-h / tc}$$

$$1,19\ Wi - 0,1\ Wi = 10,44\ \text{kw-h / tc}$$

donc Wi = 8,85

9. CONCLUSIONS

À partir des résultats obtenus lors de l'usinage de 9 005 tonnes courtes de minerai provenant du projet Fenelon, on peu conclure les éléments suivants :

- Le personnel de l'usine Camflo a travaillé de façon professionnelle tout au long de l'usinage du lot de minerai
- Le cuivre contenu dans le minerai a certainement contribué à la consommation de cyanure mais n'a causé que peu de problème lors du raffinage.
- Le graphite ne semble pas avoir causé de problème de récupération.

À partir des mêmes résultats obtenus lors de l'usinage, on peu supposer les éléments suivants :

- Le minerai riche représente 6 354 tonnes courtes d'une teneur d'environ 0,362 once par tonne courte
- Le minerai pauvre représente 2 651 tonnes courtes d'une teneur d'environ 0,148 once par tonne courte
- Le problème de pression dans les presses peu provenir de la formation d'hydroxyde de zinc. Ce problème peut être contré par une augmentation de la quantité de plomb ajouté avec la poudre de zinc.
- L'ajout de produit dans le clarificateur tel le désincrustant MILL-SPERS 805 a un effet nuisible sur la pression dans les presses.

À partir des mêmes résultats obtenus lors de l'usinage, il n'est pas possible de conclure au sujet des éléments suivants :

- Il n'est pas possible de conclure au sujet de l'efficacité de l'ajout d'oxygène dans l'agitateur No 2 sur la récupération de l'or.
- Il n'est pas possible de conclure que l'augmentation du taux de cyanure dans le circuit a eu un effet quelconque à l'exception d'augmenter la consommation.
- Il n'est pas possible d'expliqué les pertes d'or qu'il y a eu au début de l'usinage car les rapports d'opération aux solutions ne présente presque pas de perte. Il y a trois possibilités. La première serait un certain laxisme des opérateurs qui n'aurait pas vraiment fait leur test. La seconde serait que les réactifs utilisés pour fait les tests auraient été périmés ou mal dosés. La troisième serait qu'il y ai eu contamination lors de la manipulation des échantillons.
- Compte tenu des problèmes rencontrés durant le traitement de ce lot, il n'est pas possible de conclure quoi que ce soit concernant l'effet de la granulométrie sur la récupération de l'or.

10. RECOMMANDATIONS

Avant de faire l'usinage d'un autre lot, il serait important de vérifier l'effet du cuivre sur la cyanuration. Ceci implique de vérifier la possibilité de flotter le cuivre avant la cyanuration ou après la cyanuration et de comparer au rendement en cyanuration directe.

Le but de flotter le cuivre avant la cyanuration est de minimiser la consommation en cyanure ainsi que d'augmenter les revenus par la vente d'un concentré de cuivre. Il est important de vérifier par la même occasion la récupération de l'or car il est possible que le concentré de cuivre entraîne beaucoup d'or qui ne sera payé qu'a 95 % par Noranda.

Le but de flotter le cuivre après cyanuration est d'augmenter les revenu par la vente du concentré de cuivre et d'augmenter la récupération de l'or car celle qui suivra le concentré aurait été perdue à la cyanuration.

Avant d'usiner un autre lot de minerai, il serait bon de s'assurer qu'il ne contient pas de graphite. S'il contient du graphite, il serait important de vérifier l'effet du graphite sur la récupération de l'or.

Lors du prochain usinage, il serait important de doubler la quantité de plomb que l'on ajoute avec la poudre de zinc et de suivre la progression de la pression dans les presses. De plus, si le minerai ne contient pas de graphite, on devrait opérer au plus bas tonnage possible afin de broyer le minerai au maximum.

L A B O R A T O I R E   L T M i n c.

International Taurus ResourcesInc.

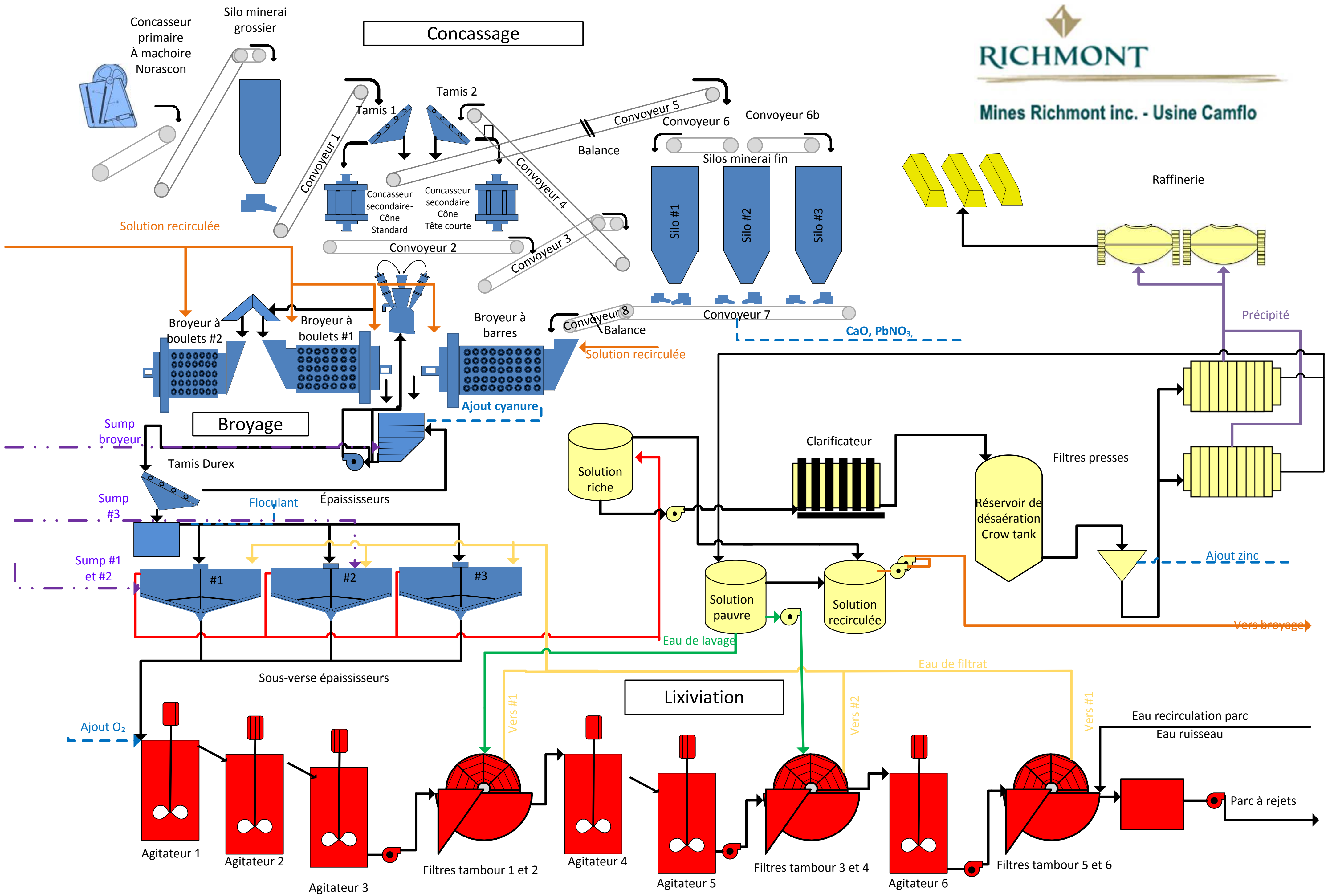
Usinage à Camflo

Projet Fenelon

RAPPORT No 1

PRÉPARÉ PAR : Edmond St-Jean ing. septembre 2004

Table des matières



RICHMONT
Mines Richmont inc. - Usine Camflo
Concassage
Concasseur primaire À machoire Norascon
Silo minerai grossier
Tamis 1
Tamis 2
Convoyeur 1
Convoyeur 2
Convoyeur 3
Convoyeur 4
Convoyeur 5
Convoyeur 6
Convoyeur 6b
Convoyeur 7
Convoyeur 8
Balance
Concasseur secondaire-Cône Standard
Concasseur secondaire Cône Tête courte
Silos minerai fin
Silo #1
Silo #2
Silo #3
Raffinerie
Solution recirculée
Broyeur à boulets #2
Broyeur à boulets #1
Broyeur à barres
CaO, PbNO3,
Ajout cyanure
Broyage
Sump broyeur
Tamis Durex
Sump #3
Floculant
Épaississeurs
Sump #1 et #2
#1
#2
#3
Précipité
Clarificateur
Filtres presses
Solution riche
Réservoir de désaération Crow tank
Ajout zinc
Solution pauvre
Solution recirculée
Vers broyage
Eau de lavage
Eau de filtrat
Sous-verse épaississeurs
Lixiviation
Ajout O2
Vers #1
Vers #2
Vers #1
Eau recirculation parc
Eau ruisseau
Parc à rejets
Agitateur 1
Agitateur 2
Agitateur 3
Filtres tambour 1 et 2
Agitateur 4
Agitateur 5
Filtres tambour 3 et 4
Agitateur 6
Filtres tambour 5 et 6

APPENDIX IV – RESERVES DATA SHEET

WALLBRIDGE MINING COMPANY LIMITED

Date: December 13, 2016
Revision: B00

SNC·LAVALIN

Appendix A: Reserves Data Sheet

Description	Stope ID	Level	Zone	Density	Mined Tonnes	Mined Volume	Grade	Mined Grams	External Dilution 1.0 g/t	Mined Diluted Tonnes	External g/t Dilution	Grams of Dilution	Total Grams	Diluted Grade	Recovery	Recovered Tonnes	Grams Recovered	New Ounces
Stope	14	5165	S1	2.80	5746	2049.40	17.68	101601	15%	6607	1.00	862	102463	15.51	97%	6409	99389	3195
Stope	21	5150	6	2.81	1355	481.88	5.68	7700	15%	1559	1.00	203	7903	5.07	80%	1247	6323	203
Stope	22	5150	S1	2.81	2193	781.46	9.88	21668	15%	2521	1.00	329	21997	8.72	97%	2446	21337	686
Stope	23	5165	5	2.81	3058	1090.29	7.39	22616	15%	3517	1.00	459	23075	6.56	95%	3341	21921	705
Stope	24	5165	S1	2.81	571	203.56	5.11	2919	15%	657	1.00	86	3005	4.57	80%	526	2404	77
Stope	25	5180	4&5	2.84	6917	2434.08	11.41	78916	15%	7955	1.00	1038	79953	10.05	97%	7716	77555	2493
Stope	26	5180	6	2.81	4408	1570.85	11.57	51020	15%	5069	1.00	661	51681	10.19	97%	4917	50131	1612
Stope	27	5150	S1	2.80	3306	1178.73	20.53	67861	15%	3801	1.00	496	68356	17.98	97%	3687	66306	2132
Stope	28	5180	5	2.81	1041	370.36	6.55	6814	15%	1197	1.00	156	6970	5.82	80%	958	5576	179
Stope	31	5195	6	2.80	7106	2533.40	10.89	77357	15%	8172	1.00	1066	78423	9.60	90%	7355	70580	2269
Stope	32	5195	S1	2.80	4283	1527.64	8.49	36364	15%	4925	1.00	642	37006	7.51	97%	4777	35896	1154
Stope	33	5195	4	2.81	2603	926.17	7.72	20097	15%	2993	1.00	390	20487	6.84	97%	2903	19873	639
Stope	34	5195	5	2.81	1084	386.19	6.84	7419	15%	1246	1.00	163	7581	6.08	95%	1184	7202	232
Stope	35	5180	S1	2.81	2408	857.64	5.32	12815	15%	2770	1.00	361	13176	4.76	95%	2631	12517	402
Stope	36	5195	6	2.81	5511	1964.06	13.31	73371	15%	6338	1.00	827	74197	11.71	90%	5704	66778	2147
Stope	37	5210	5	2.81	1667	593.13	5.02	8376	15%	1917	1.00	250	8626	4.50	97%	1860	8368	269
Stope	38	5210	6	2.81	2065	735.89	10.20	21067	15%	2375	1.00	310	21377	9.00	97%	2304	20736	667
Stope	39	5210	6	2.81	1357	483.38	14.36	19477	15%	1560	1.00	203	19681	12.62	97%	1513	19090	614
Stope	40	5210	6	2.81	923	328.85	9.42	8697	15%	1062	1.00	138	8835	8.32	80%	849	7068	227
Stope	41	5210	5	2.81	6527	2326.84	12.55	81887	15%	7506	1.00	979	82866	11.04	97%	7281	80380	2584
Stope	42	5210	3	2.81	3346	1191.00	10.81	36183	15%	3848	1.00	502	36685	9.53	85%	3271	31182	1003
Stope	43	5210	6	2.81	1624	578.78	7.72	12543	15%	1868	1.00	244	12787	6.85	80%	1494	10229	329
Stope	44	5225	3	2.81	3574	1274.28	5.73	20470	15%	4111	1.00	536	21006	5.11	95%	3905	19956	642
Stope	45	5180	6	2.81	766	272.28	11.77	9017	15%	881	1.00	115	9132	10.37	97%	854	8858	285
Stope	46	5195	3	2.81	1502	534.73	5.61	8422	15%	1727	1.00	225	8647	5.01	90%	1555	7782	250
Stope	47	5195	3	2.81	1551	552.63	10.19	15797	15%	1784	1.00	233	16030	8.99	85%	1516	13625	438
Pillar	48	5150	S1	2.80	1180	421.09	13.11	15469	15%	1357	1.00	177	15646	11.53	97%	1317	15176	488
Pillar	49	5180	6	2.81	574	204.09	8.85	5086	15%	661	1.00	86	5172	7.83	97%	641	5017	161
Stope	50	Pit	4	2.80	1557	556.52	11.24	17502	15%	1790	1.00	234	17736	9.91	97%	1737	17204	553
Stope	51	Pit	3	2.80	354	126.36	6.59	2334	15%	407	1.00	53	2387	5.86	97%	395	2316	74
Stope	52	Pit	2	2.75	311	112.85	5.72	1777	15%	357	1.00	47	1824	5.11	97%	346	1769	57
Pillar	56	5195	6	2.80	3026	1079.40	6.11	18488	15%	3480	1.00	454	18942	5.44	95%	3306	17995	579
Drift & Fill	-	5150	S1	2.81	167	59.56	11.00	1842	5%	176	1.00	8	1850	10.53	97%	171	1795	58
Drift & Fill	-	5150	S1	2.81	72	25.83	5.12	371	5%	76	1.00	4	374	4.92	97%	74	363	12
Drift & Fill	-	5150	S1	2.81	158	56.19	9.44	1493	5%	166	1.00	8	1501	9.04	97%	161	1456	47
Drift & Fill	-	5165	S1	2.80	69	24.80	14.01	973	5%	73	1.00	3	977	13.39	97%	71	947	30
Drift & Fill	-	5165	S1	2.81	182	64.68	13.54	2458	5%	191	1.00	9	2467	12.94	97%	185	2393	77
Drift & Fill	-	5180	4	2.82	182	64.74	8.58	1564	5%	191	1.00	9	1573	8.22	97%	186	1526	49
Drift & Fill	-	5180	4	2.81	195	69.42	9.19	1794	5%	205	1.00	10	1804	8.80	97%	199	1750	56
Drift & Fill	-	5195	5	2.80	117	41.92	5.33	626	5%	123	1.00	6	632	5.13	97%	120	613	20

Description	Stope ID	Level	Zone	Density	Mined Tonnes	Mined Volume	Grade	Mined Grams	External Dilution 1.0 g/t	Mined Diluted Tonnes	External g/t Dilution	Grams of Dilution	Total Grams	Diluted Grade	Recovery	Recovered Tonnes	Grams Recovered	New Ounces
Drift & Fill	-	5195	5	2.80	172	61.32	5.33	915	5%	180	1.00	9	923	5.12	97%	175	896	29
Drift & Fill	-	5195	4	2.82	164	58.26	5.85	961	5%	173	1.00	8	970	5.62	97%	167	941	30
Drift & Fill	-	5195	4	2.82	166	58.99	6.01	999	5%	175	1.00	8	1007	5.77	97%	169	977	31
Drift & Fill	-	5210	5	2.80	165	59.01	11.35	1875	5%	174	1.00	8	1884	10.86	97%	168	1827	59
Drift & Fill	-	5210	5	2.81	88	31.21	6.47	567	5%	92	1.00	4	571	6.21	97%	89	554	18
Drift & Fill	-	5210	4	2.80	333	118.86	9.33	3110	5%	350	1.00	17	3127	8.93	97%	340	3033	98
Sill	-	5180	5	2.82	92	32.63	6.49	597	5%	97	1.00	5	602	6.23	97%	94	584	19
Sill	-	5180	S1	2.80	87	31.03	9.45	822	5%	91	1.00	4	826	9.05	97%	89	801	26
Sill	-	5180	S1	2.80	179	63.97	6.87	1231	5%	188	1.00	9	1240	6.59	97%	182	1203	39
Sill	-	5210	S1	2.82	81	28.52	7.08	571	5%	85	1.00	4	575	6.79	97%	82	557	18
Sill	-	5210	S1	2.82	81	28.62	9.58	773	5%	85	1.00	4	777	9.17	97%	82	753	24
Sill	-	5210	4	2.81	390	138.83	8.40	3279	5%	410	1.00	20	3298	8.05	97%	398	3199	103
Sill	-	5165	S1	2.81	72	25.75	6.25	452	5%	76	1.00	4	455	6.00	97%	74	442	14
Sill	-	5165	S1	2.82	174	61.53	10.35	1797	5%	182	1.00	9	1806	9.91	97%	177	1752	56
Sill	-	5210	5	2.81	68	24.10	6.74	457	5%	71	1.00	3	460	6.47	97%	69	446	14
Sill	-	5210	5	2.81	119	42.42	8.95	1066	5%	125	1.00	6	1072	8.57	97%	121	1040	33
Sill	-	5210	5	2.82	124	44.12	5.95	740	5%	131	1.00	6	746	5.72	97%	127	724	23
Broken Ore	-	5210	-	-	3100	-	6.14	19034	0%	3100	0.00	0	19034	6.14	97%	3007	18463	594
Total													**954201**	**9.30**		**96721**	**899570**	**28922**

APPENDIX V – MINE AND INFRASTRUCTURE DRAWINGS

Appendix B

Mine and Infrastructure Drawings

Drawing Number	Title
640914-0000-45DD-0001-B00	Surface – Infrastructures – Site – General Arrangement
640914-0000-45DD-0002-B00	Surface – Infrastructures – Site – General Arrangement
640914-0000-45DD-0003-B00	Surface – Infrastructures – Site – General Arrangement
640914-0000-45DD-0100-B00	Underground – Infrastructures – Small Comfort Station – Typical – General Arrangement
640914-0000-45DD-0101-B00	Underground – Infrastructures – Portable Refuge Station – Typical – General Arrangement
640914-0000-45DD-0102-B00	Underground – Infrastructures – Blasting Cap Storage – Typical – General Arrangement
640914-0000-45DD-0103-B00	Underground – Infrastructures – Explosive Storage – Typical – General Arrangement
640914-0000-45DD-0200-B00	Surface & Underground – Infrastructures – Water Distribution Pump System – Flowsheet
640914-0000-45DD-0201-B00	Surface & Underground – Infrastructures – Compressed Air Distribution System – Flow Diagram
Figure 1A	Settling Basin Option 1
Figure 1B	Settling Basin Option 2
640914-0000-47D1-0001_B00	Électricité – Wallbridge Mining – Fenelon Project – Surface Distribution – Underground Distribution – Single Line Diagram
640914-0000-47EL-0001_B00	Fenelon Gold Project – Surface Load List
640914-0000-4MDD-0001_B00	Underground Infrastructure – Development & Production Phases – Ramp – Sections
640914-0000-4MDD-0002_B00	Underground Infrastructure – Production Phase – Level Access & Sill – Sections
640914-0000-4MDD-0003_B00	Underground Infrastructure – Ventilation Bulkhead – Typical – Plan & Section
640914-0000-4MDD-0004_B00	Underground Mine Design – Typical Layout – 5150 Level – Plan
640914-0000-4MDD-0005_B00	Underground Mine Design – Typical Layout – 5165 Level – Plan
640914-0000-4MDD-0006_B00	Underground Mine Design – Typical Layout – 5180 Level – Plan
640914-0000-4MDD-0007_B00	Underground Mine Design – Typical Layout – 5195 Level – Plan
640914-0000-4MDD-0008_B00	Underground Mine Design – Typical Layout – 5210 Level – Plan
640914-0000-4MDD-0009_B00	Underground Mine Design – Typical Layout – 5225 Level – Plan

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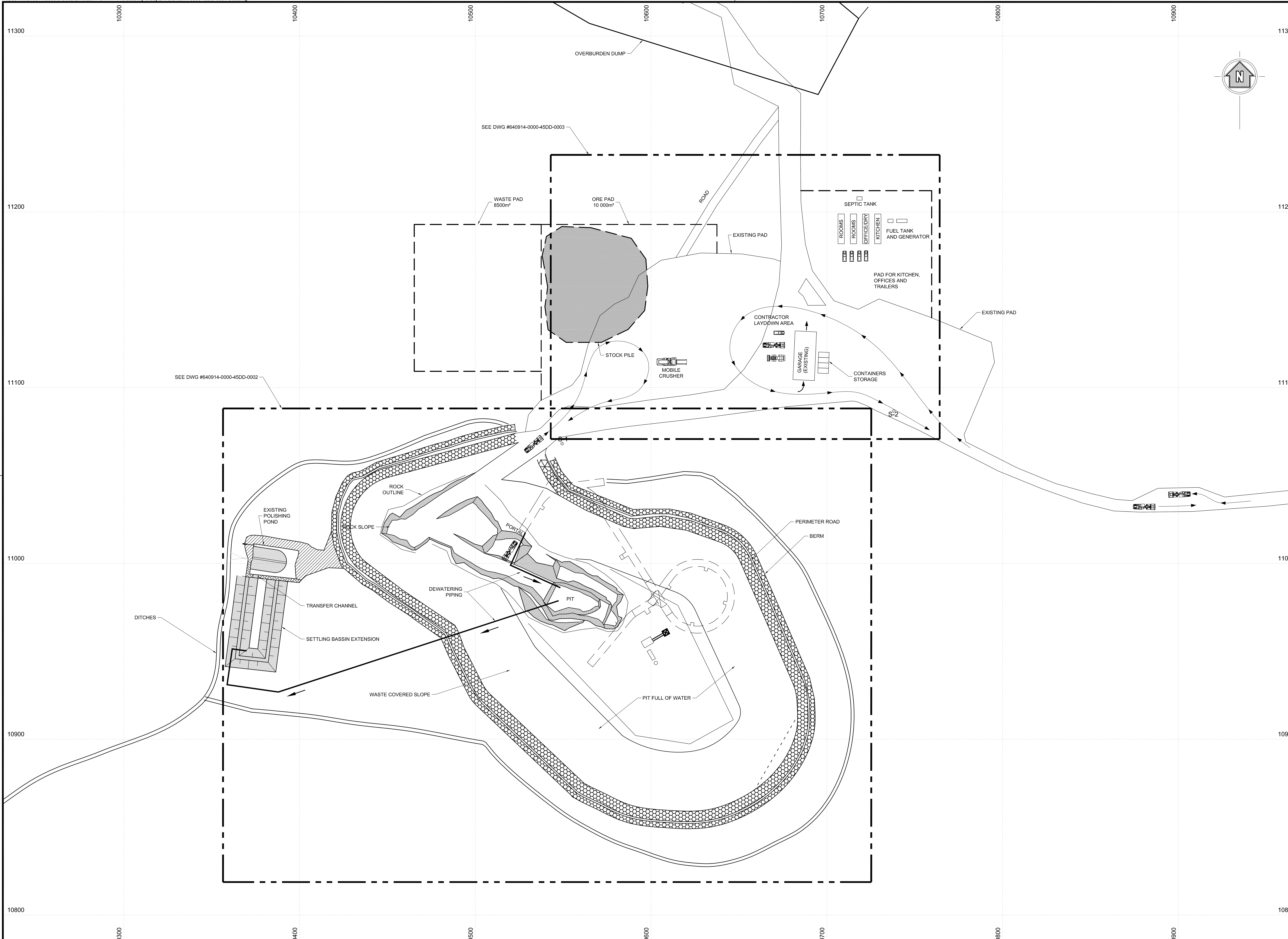

Client :

WALLBRIDGE
MINING COMPANY LIMITED

PRÉLIMINAIRE
NE PAS UTILISER POUR LA CONSTRUCTION
PRELIMINARY
DO NOT USE FOR CONSTRUCTION
SNC·LAVALIN
2017-01-11

REV.	DATE	ISSUES	BY	CHK	APP.
B00	2017-01-11	ISSUED FOR PREFEASIBILITY STUDY	B.LEM.	M.HÉN.	-
BPC	2016-12-14	ISSUED FOR FINAL REVIEW	B.LEM.	M.HÉN.	-
B	2016-11-11	ISSUED FOR COMMENTS	B.LEM.	M.HÉN.	-
A	2016-11-01	ISSUED FOR COMMENTS	B.LEM.	M.HÉN.	-

Seal(s) :

SNC·LAVALIN
SNC-Lavalin Stavibel inc.
150, rue Gamble Ouest
Rouyn-Noranda (Québec) J9X 2R7
Tel. : 819 764-5181
Fax : 819 797-0158
www.snclavalin.com

Project :

FENELON STUDY

Title :

SURFACE
INFRASTRUCTURES
SITE
-
-
GENERAL ARRANGEMENT

Drawn by : Bruno Lemieux, tech.	Org. Date : 2016-10-25
Designed by : Marc Hénault, P. eng.	Project manager : Marc Hénault, P. eng.
Verified by : Marc Hénault, P. eng.	Project # : 640914-0000
Approved by : Marc Hénault, P. eng.	Reference : -
Scale : 1 : 1000	Drawing # : 640914-0000-45DD-0001 — Rev. : B00

Tracé par: "Lemib", 2017-01-11 10:19:21 (Lts: 1.00 ; Mea: 1)

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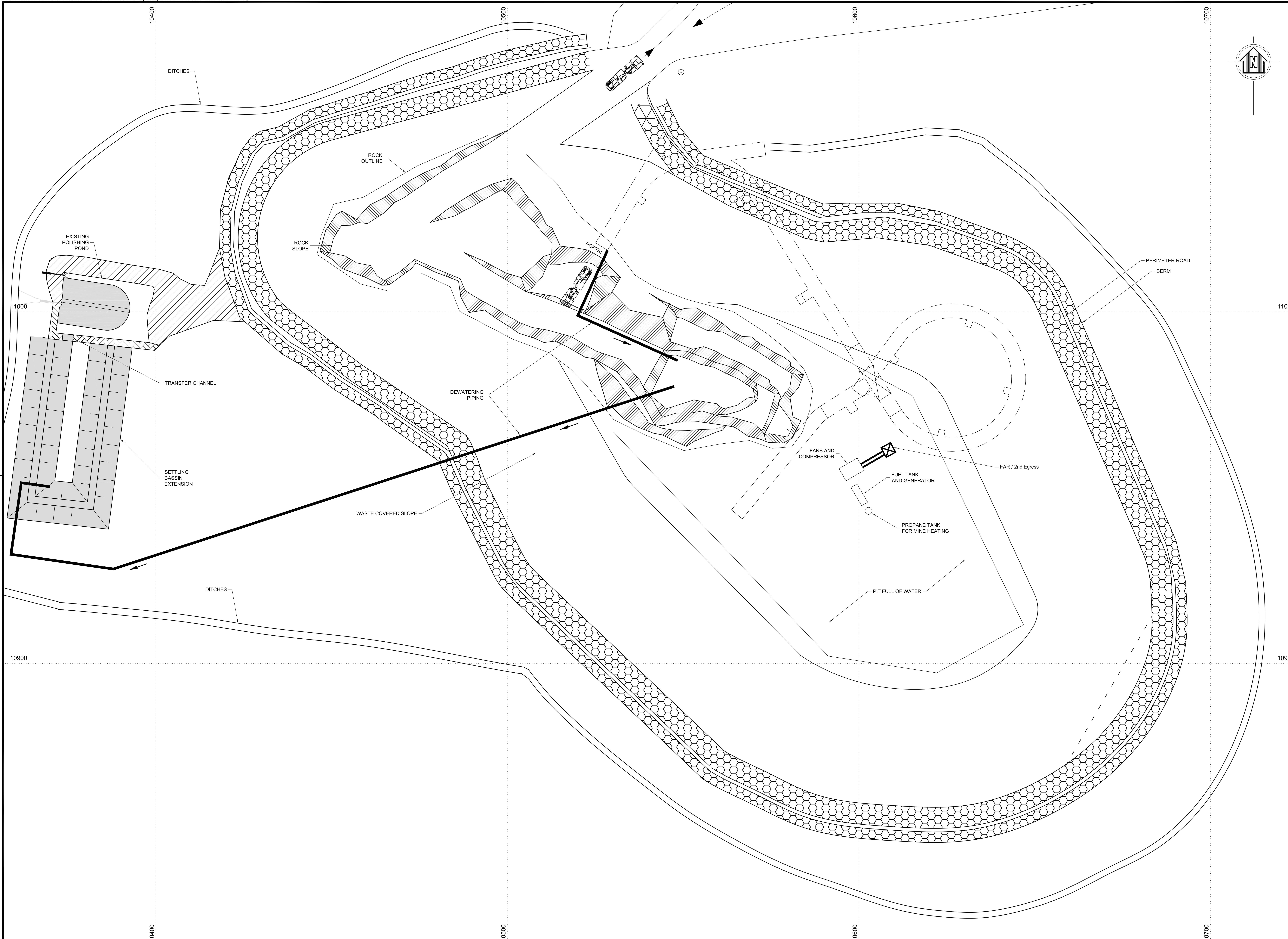

Client : WALLBRIDGE MINING COMPANY LIMITED

REV.	DATE	ISSUES	BY	CHK	APP.
B00	2017-01-11	ISSUED FOR PREFEASIBILITY STUDY	B.LEM.	M.HÉN.	-
BPC	2016-12-14	ISSUED FOR FINAL REVIEW	B.LEM.	M.HÉN.	-
B	2016-11-11	ISSUED FOR COMMENTS	B.LEM.	M.HÉN.	-
A	2016-11-01	ISSUED FOR COMMENTS	B.LEM.	M.HÉN.	-

Seal(s) :

SNC·LAVALIN

SNC-Lavalin Stavibel inc.
150, rue Gamble Ouest
Rouyn-Noranda (Québec) J9X 2R7
Tel. : 819 764-5181
Fax : 819 797-0158
www.snclavalin.com

Project :

FENELON STUDY

Title :

SURFACE
INFRASTRUCTURES
SITE
-
-
GENERAL ARRANGEMENT

Drawn by : Bruno Lemieux, tech.	Org. Date : 2016-10-25
Designed by : Marc Hénault, P. eng.	Project manager : Marc Hénault, P. eng.
Verified by : Marc Hénault, P. eng.	Project # : 640914-0000
Approved by : Marc Hénault, P. eng.	Reference : -
Scale : 1 : 500	Drawing # : 640914-0000-45DD-0002
Rev. : B00	

Tracé par: "Lemib", 2017-01-11 10:21:32 (Lts: 1.00 ; Mea: 1)

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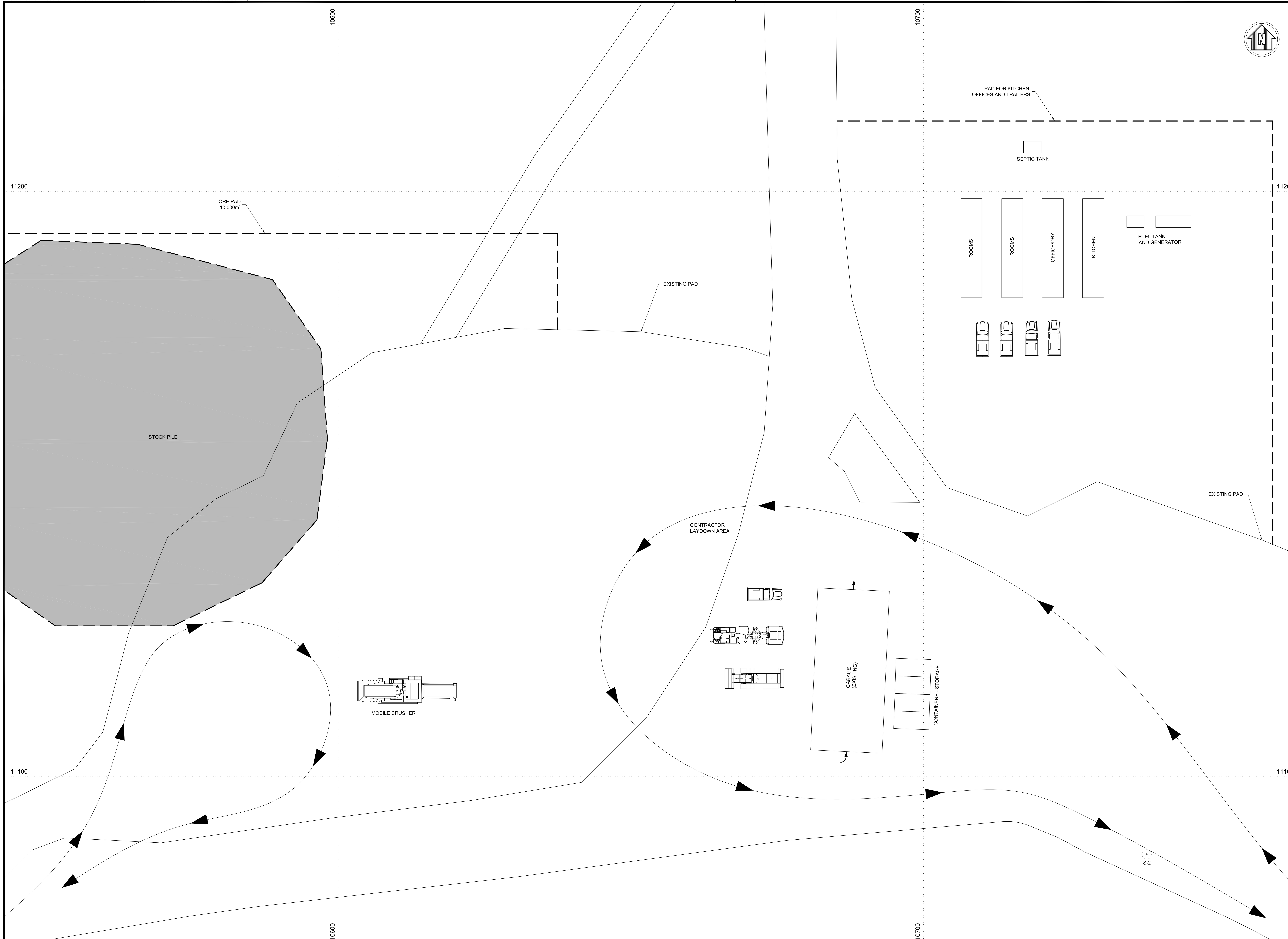

Client :

WALLBRIDGE MINING COMPANY LIMITED

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SNC·LAVALIN
2017-01-11

REV.	DATE	ISSUES	BY	CHK	APP.
B00	2017-01-11	ISSUED FOR PREFEASIBILITY STUDY	B.LEM.	M.HÉN.	-
BPB	2016-12-14	ISSUED FOR FINAL REVIEW	B.LEM.	M.HÉN.	-
B	2016-11-11	ISSUED FOR COMMENTS	B.LEM.	M.HÉN.	-
A	2016-11-01	ISSUED FOR COMMENTS	B.LEM.	M.HÉN.	-

Seal(s) :

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Tel. : 819 764-5181
Fax : 819 797-0158
www.snclavalin.com

Project :

FENELON STUDY

Title :

SURFACE INFRASTRUCTURES
SITE
-
-
GENERAL ARRANGEMENT

Drawn by : Bruno Lemieux, tech.	Org. Date : 2016-10-25
Designed by : Marc Hénault, P. eng.	Project manager : Marc Hénault, P. eng.
Verified by : Marc Hénault, P. eng.	Project # : 640914-0000
Approved by : Marc Hénault, P. eng.	Reference : -
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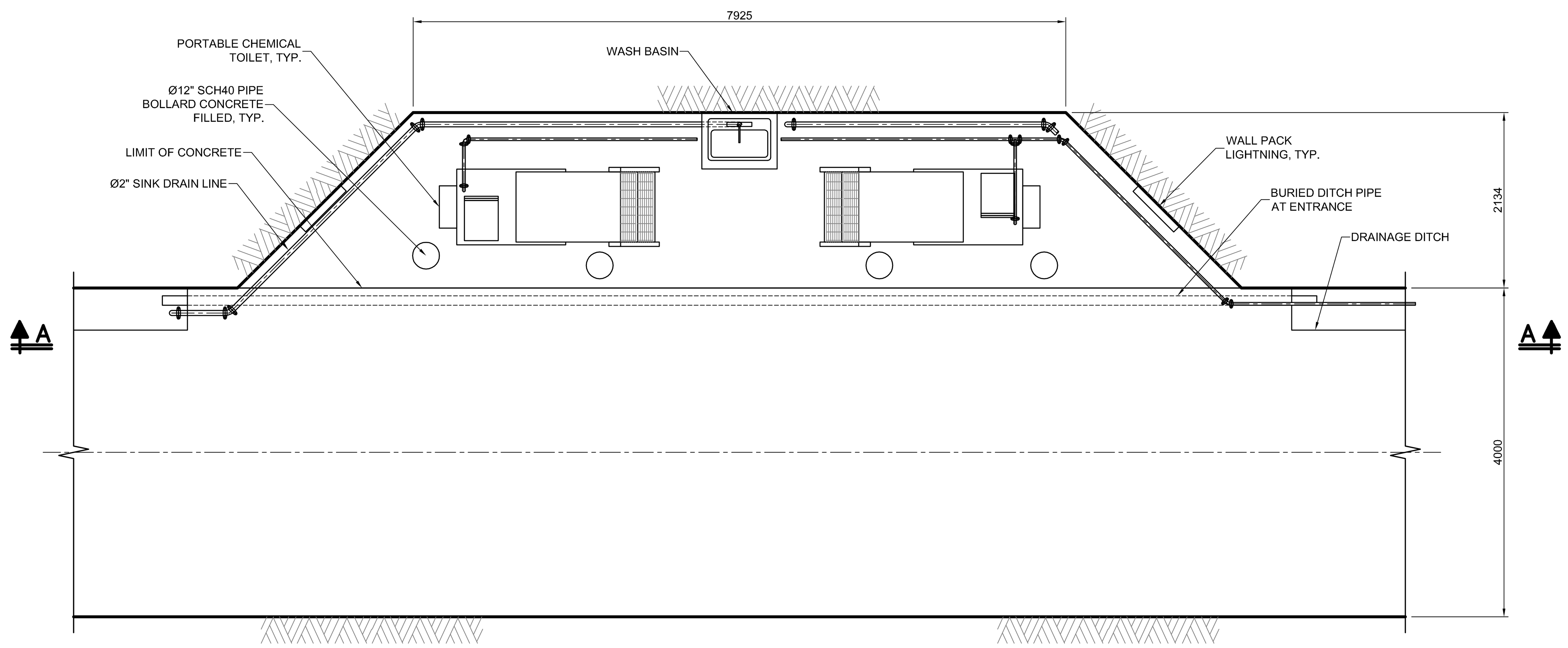
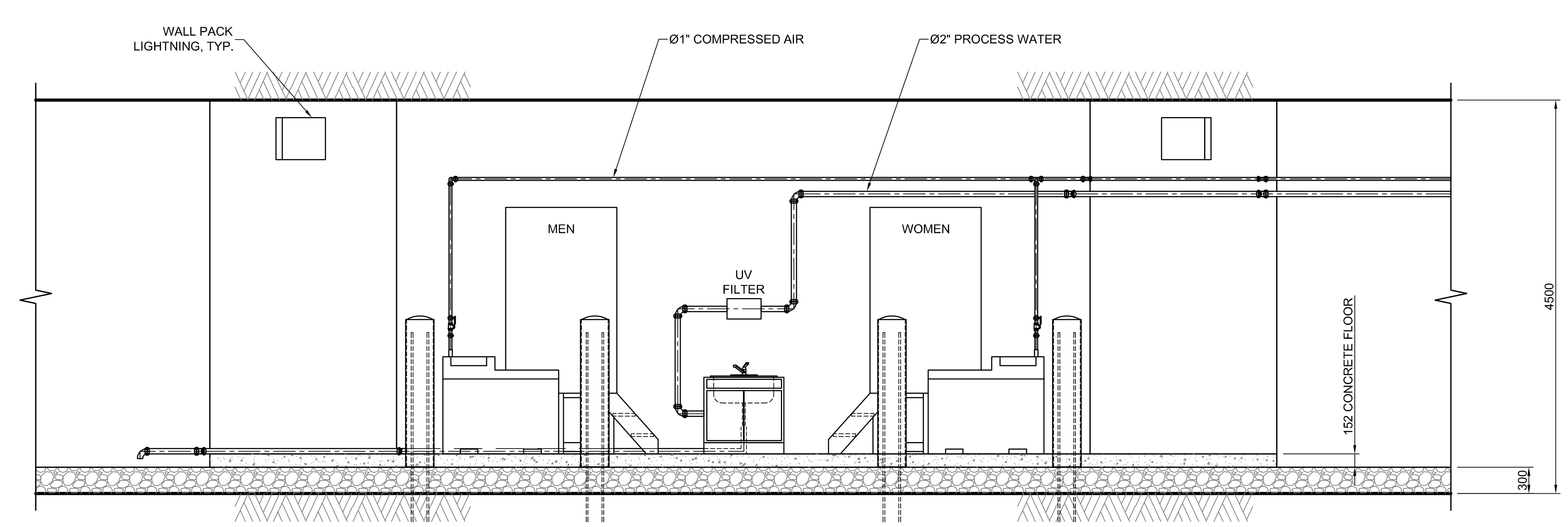


PLAN VIEW
SCALE : 1 : 50



SECTION A-A
SCALE : 1 : 50

NOTES:

1. ONE MINING TOILET IS REQUIRED FOR EVERY 25 WORKERS
2. LIGHTNING IS TO BE PROVIDED FOR SMALL COMFORT STATION.
3. DRIFT DIMENSIONS MAY VARY DEPENDING ON LOCATIONS.
4. 2" SHOTCRETE LINING ON ALL WALL AND BACK.

Client :

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2017-01-11

REV.	DATE	ISSUES	BY	CHK	APP.
B00	2017-01-11	ISSUED FOR PREFEASIBILITY STUDY	B.LEM.	M.HÉN.	-
BPB	2016-12-14	ISSUED FOR FINAL REVIEW	B.LEM.	M.HÉN.	-
B	2016-11-11	ISSUED FOR COMMENTS	B.LEM.	M.HÉN.	-
A	2016-11-01	ISSUED FOR COMMENTS	B.LEM.	M.HÉN.	-

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150, rue Gamble Ouest
Rouyn-Noranda (Québec) J9X 2R7
Tel. : 819 764-5181
Fax : 819 797-0158
www.snclavalin.com

Project :

FENELON STUDY

Title :

UNDERGROUND
INFRASTRUCTURES
SMALL CONFORT STATION
-
TYPICAL
GENERAL ARRANGEMENT

Drawn by : Bruno Lemieux, tech.	Org. Date : 2016-10-25
Designed by : Marc Hénault, P. eng.	Project manager : Marc Hénault, P. eng.
Verified by : Marc Hénault, P. eng.	Project # : 640914-0000
Approved by : Marc Hénault, P. eng.	Reference : -
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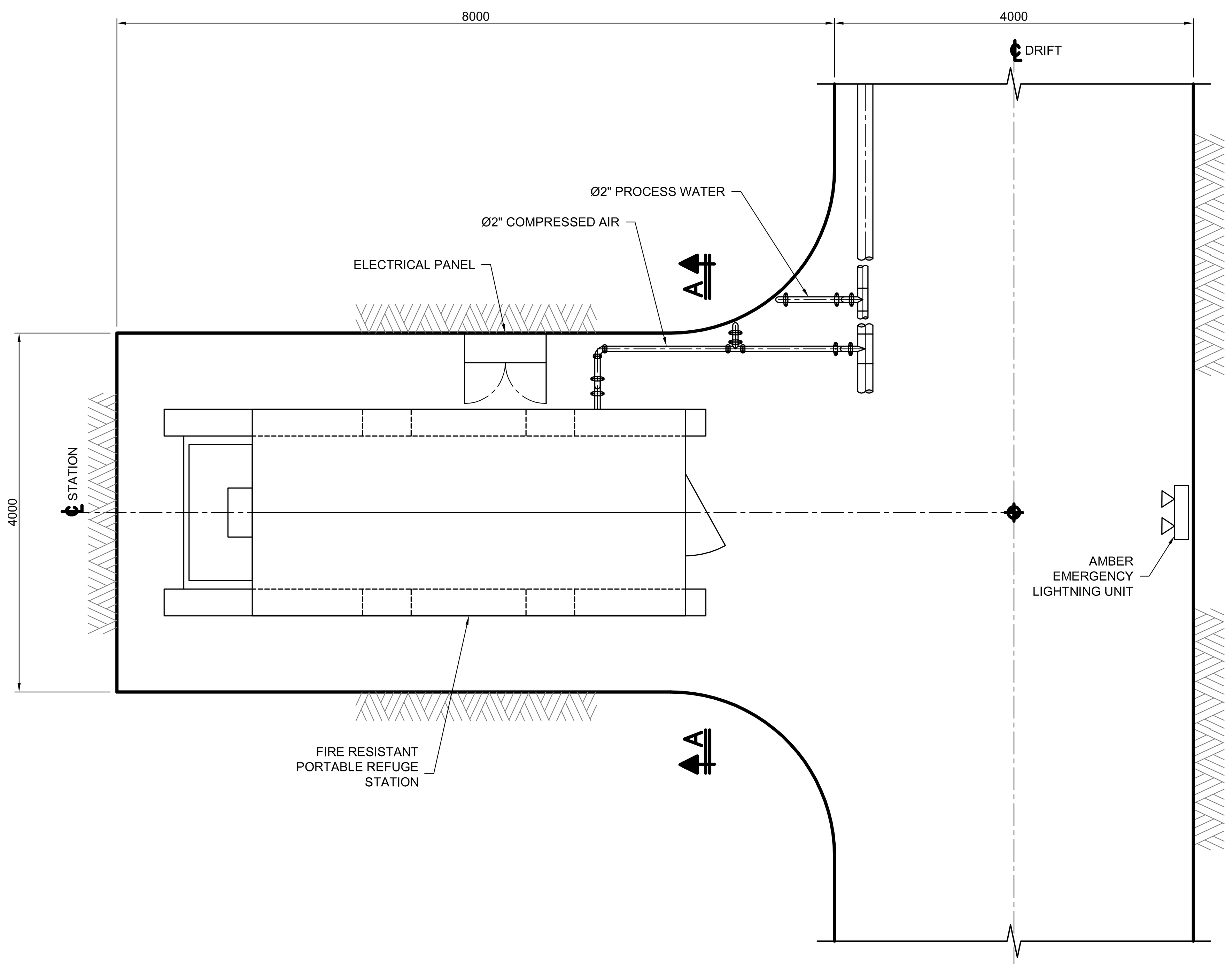
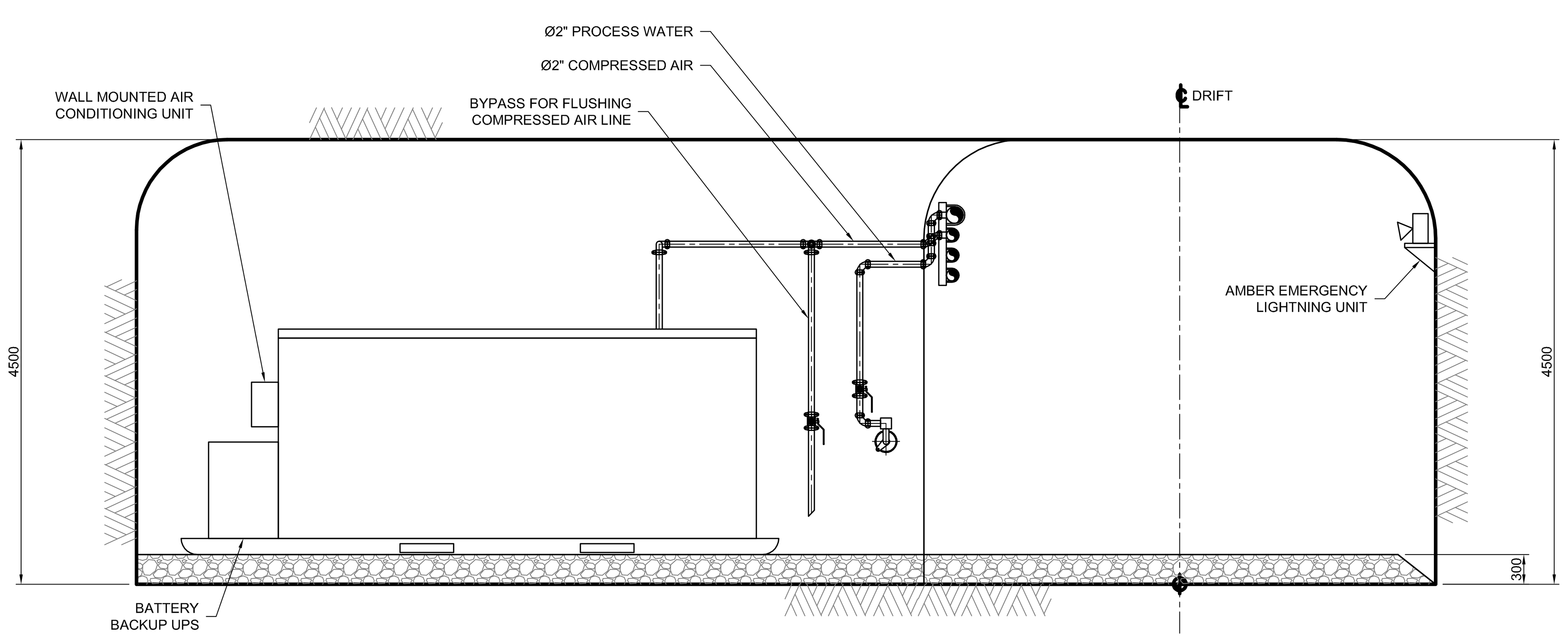


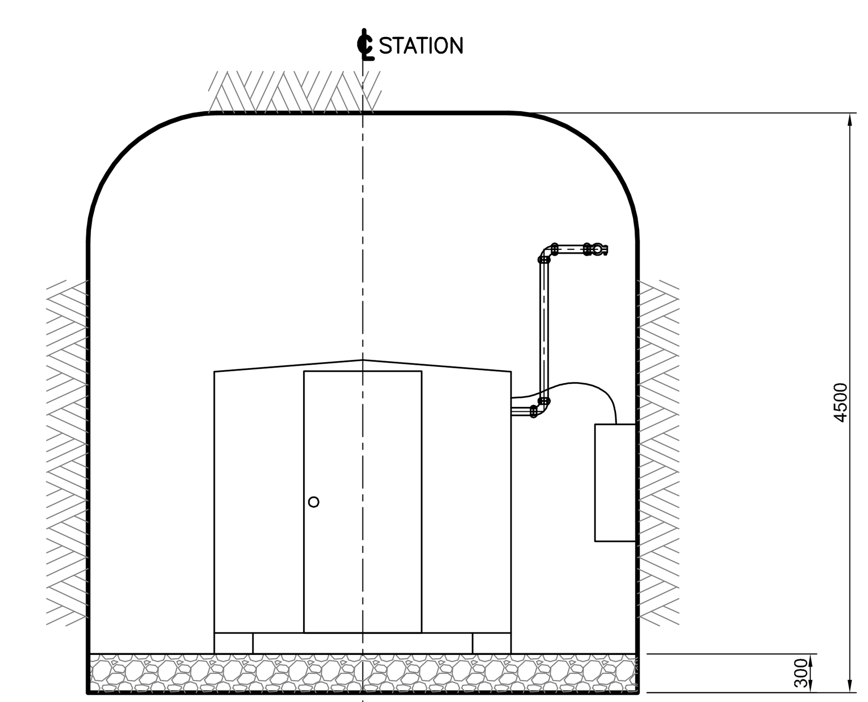
PLAN VIEW

SCALE : 1 : 50



ELEVATION

SCALE : 1 : 50



SECTION A-A

SCALE : 1 : 50

PORTABLE REFUGE STATION SERVICES

ITEM	DESCRIPTION
BENCHES	INSTALLED ALONG TWO WALLS.
DOORS	DOUBLE DOOR AIR PRESSURE LOCKS AND ESCAPE HATCH (REAR OF STATION), C/W BOTTOM SILL, ADJUSTABLE EXHAUST PORT AND FIRE RESISTANT DOOR SEALS.
FLOOR	NON-SLIP RUBBER FLOOR.
OXYGEN REPLENISHING AIR SYSTEM (ORAS)	USED TO REMOVE CO_2 FROM AIR. OXYGEN SUPPLEMENTATION VIA STANDARD PRESSURIZED OXYGEN CONNECTION TO OXYGEN BOTTLES.
AIR CONDITIONNER	LOCATED AT REAR OF STATION TO REDUCE THE TEMPERATURE, ACTS AS A DEHUMIDIFIER.
LIGHTNING	PERMANENT LIGHTNING IN THE REFUGE STATION, OPERATES FROM BOTH BATTERY AND MINE POWER.
TOILET	SANITARY CHEMICAL TOILET WITH WASTE TANK CAPACITY OF 14L AND HOLDING TANK CAPACITY OF 24L (EMERGENCY USE ONLY).
WATER	STORAGE FOR WATER BOTTLES UNDER BENCHES (MINIMUM 12 LITRES).
COMPRESSED AIR LINE CONNECTION AND FILTER	COMPRESSED AIR LINE C/W 3 STAGE FILTERS & SILENCER MOUNTED IN THE MAIN CHAMBER, PRESSURE RELIEF VALVES MOUNTED ON THE EXTERIOR.
SAFETY EQUIPEMENT	STRETCHER, FIRST AID KIT, BLANKETS AND TOOL KIT
FIRE EXTINGUISHER	10 LB ABC TYPE
DOCUMENTS	LEVEL PLAN DRAWING, FIRE PROCEDURE, REFUGE STATION PROCEDURE.
COMMUNICATIONS	TELEPHONE, LIST OF TELEPHONE NUMBERS
BATTERY BACKUP UPS	MINIMUM 36 HR CAPACITY TO POWER REFUGE'S INTERNAL LIFE SUPPORT SYSTEMS.

NOTES:
1. UNIT CAN BROKEN DOWN FOR TRANSPORT.
2. HAND WASHING FACILITIES ARE LOCATED IN NEARBY COMFORT STATION.
3. DRIFT DIMENSIONS MAY VARY DEPENDING ON LOCATION.
4. ALL EXTERNAL SERVICE CONNECTIONS TO THE REFUGE STATION (COMPRESSED AIR, POWER, COMMUNICATIONS) TO BE SEALED WITH FIRE RESISTANT MATERIALS.

Client :

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2017-01-11

REV.	DATE	ISSUES	BY	CHK	APP.
B00	2017-01-11	ISSUED FOR PREFEASIBILITY STUDY	B.LEM.	M.HÉN.	-
BPB	2016-12-14	ISSUED FOR FINAL REVIEW	B.LEM.	M.HÉN.	-
B	2016-11-11	ISSUED FOR COMMENTS	B.LEM.	M.HÉN.	-
A	2016-11-01	ISSUED FOR COMMENTS	B.LEM.	M.HÉN.	-

Seal(s) :

SNC·LAVALIN
SNC-Lavalin Stavibel inc.
150, rue Gamble Ouest
Rouyn-Noranda (Québec) J9X 2R7
Tel. : 819 764-5181
Fax : 819 797-0158
www.snclavalin.com

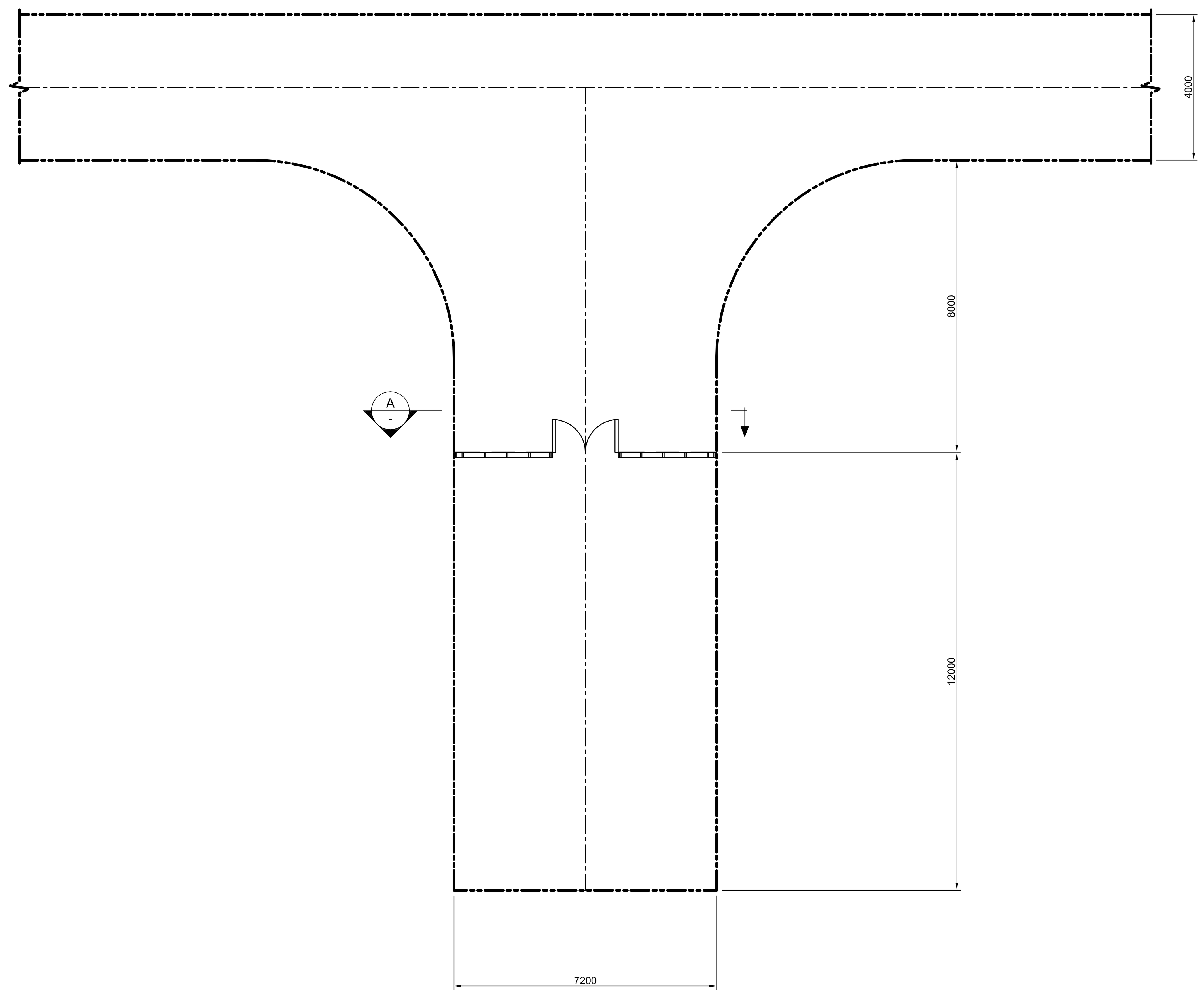
Project :

FENELON STUDY

Title :

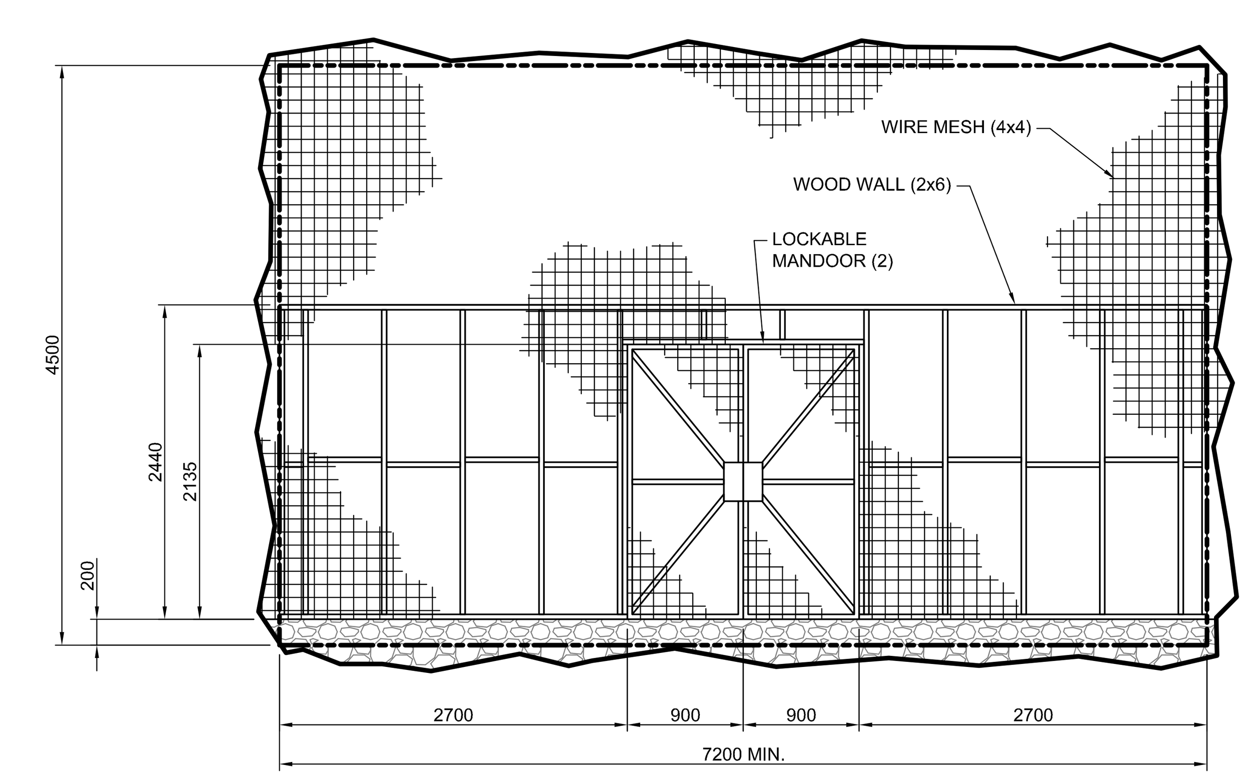
UNDERGROUND
INFRASTRUCTURES
PORTABLE REFUGE STATION
-
TYPICAL
GENERAL ARRANGEMENT

Drawn by : Bruno Lemieux, tech.	Org. Date : 2016-10-25
Designed by : Marc Hénault, P. eng.	Project manager : Marc Hénault, P. eng.
Verified by : Marc Hénault, P. eng.	Project # : 640914-0000
Approved by : Marc Hénault, P. eng.	Reference : -
Scale : 1 : 50	Drawing # : 640914-0000-45DD-0101 — Rev. : B00



PLAN VIEW

SCALE : 1 : 100



SECTION A -

SCALE : 1 : 50

Client :

WALLBRIDGE MINING COMPANY LIMITED

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2017-01-11

REV.	DATE	ISSUES	BY	CHK	APP.
B00	2017-01-11	ISSUED FOR PREFEASIBILITY STUDY	B.LEM.	M.HÉN.	-
BPC	2016-12-14	ISSUED FOR FINAL REVIEW	B.LEM.	M.HÉN.	-
C	2016-11-14	ISSUED FOR COMMENTS	B.LEM.	M.HÉN.	-
B	2016-11-11	ISSUED FOR COMMENTS	B.LEM.	M.HÉN.	-
A	2016-11-01	ISSUED FOR COMMENTS	B.LEM.	M.HÉN.	-

Seal(s) :

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150, rue Gamble Ouest
Rouyn-Noranda (Québec) J9X 2R7
Tel. : 819 764-5181
Fax : 819 797-0158
www.snclavalin.com

Project :

FENELON STUDY

Title :

UNDERGROUND
INFRASTRUCTURES
BLASTING CAP STORAGE
-
TYPICAL
GENERAL ARRANGEMENT

Drawn by : Bruno Lemieux, tech.	Org. Date : 2016-10-25
Designed by : Marc Hénault, P. eng.	Project manager : Marc Hénault, P. eng.
Verified by : Marc Hénault, P. eng.	Project # : 640914-0000
Approved by : Marc Hénault, P. eng.	Reference : -
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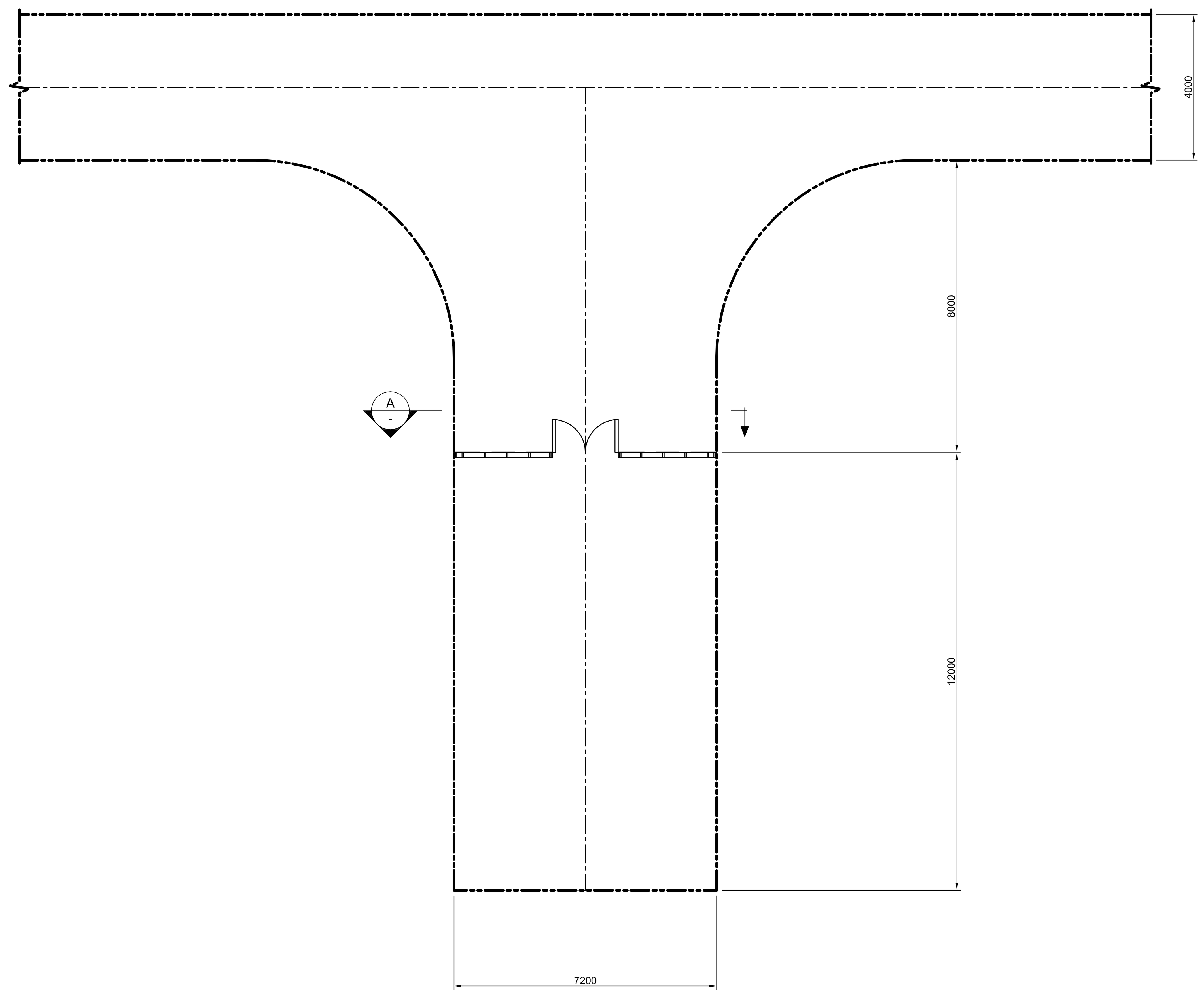
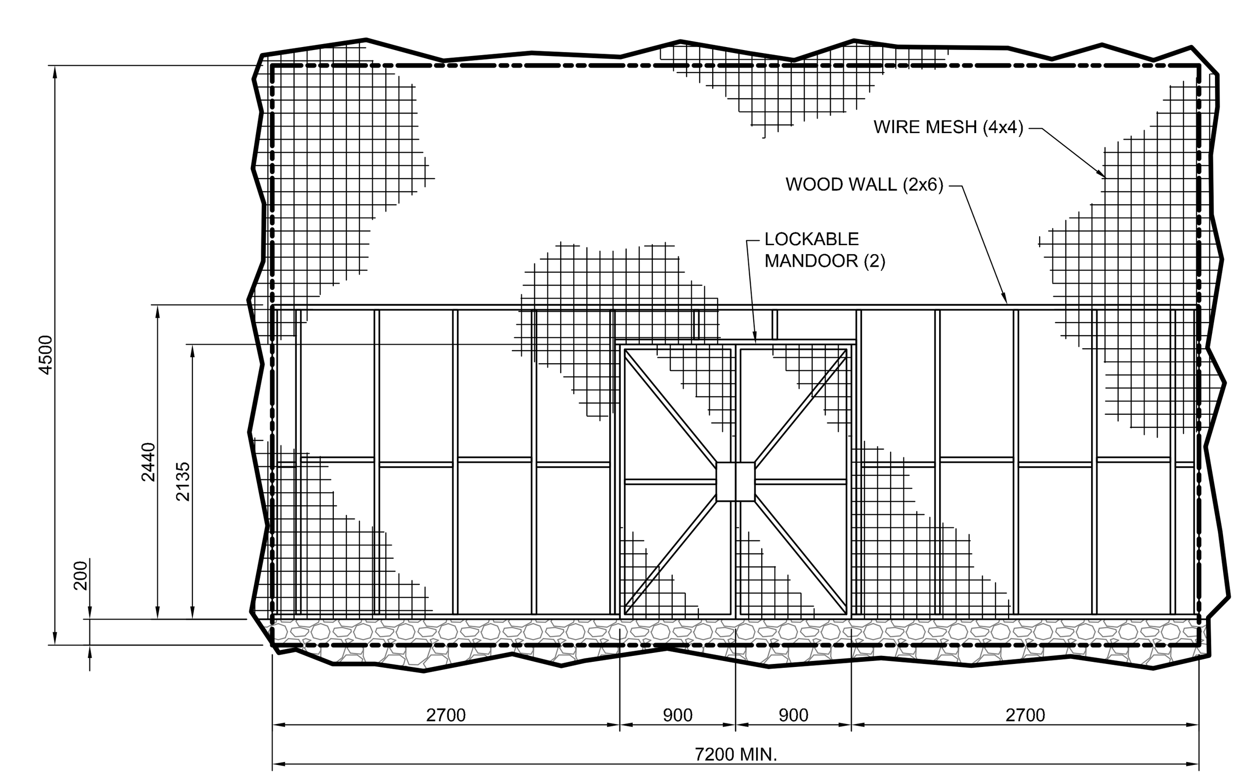


PLAN VIEW
SCALE : 1 : 100



SECTION A
SCALE : 1 : 50

Client :

WALLBRIDGE MINING COMPANY LIMITED

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PRELIMINARY
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SNC·LAVALIN
2017-01-11

REV.	DATE	ISSUES	BY	CHK	APP.
B00	2017-01-11	ISSUED FOR PREFEASIBILITY STUDY	B.LEM.	M.HÉN.	-
BPC	2016-12-14	ISSUED FOR FINAL REVIEW	B.LEM.	M.HÉN.	-
C	2016-11-14	ISSUED FOR COMMENTS	B.LEM.	M.HÉN.	-
B	2016-11-11	ISSUED FOR COMMENTS	B.LEM.	M.HÉN.	-
A	2016-11-01	ISSUED FOR COMMENTS	B.LEM.	M.HÉN.	-

Seal(s) :

SNC·LAVALIN

SNC-Lavalin Stavibel inc.
150, rue Gamble Ouest
Rouyn-Noranda (Québec) J9X 2R7
Tel. : 819 764-5181
Fax : 819 797-0158
www.snclavalin.com

Project :

FENELON STUDY

Title :

UNDERGROUND
INFRASTRUCTURES
EXPLOSIVE STORAGE
-
TYPICAL
GENERAL ARRANGEMENT

Drawn by : Bruno Lemieux, tech.	Org. Date : 2016-10-25
Designed by : Marc Hénault, P. eng.	Project manager : Marc Hénault, P. eng.
Verified by : Marc Hénault, P. eng.	Project # : 640914-0000
Approved by : Marc Hénault, P. eng.	Reference : -
Scale : INDICATED	Drawing # : 640914-0000-45DD-0103 — Rev. : B00

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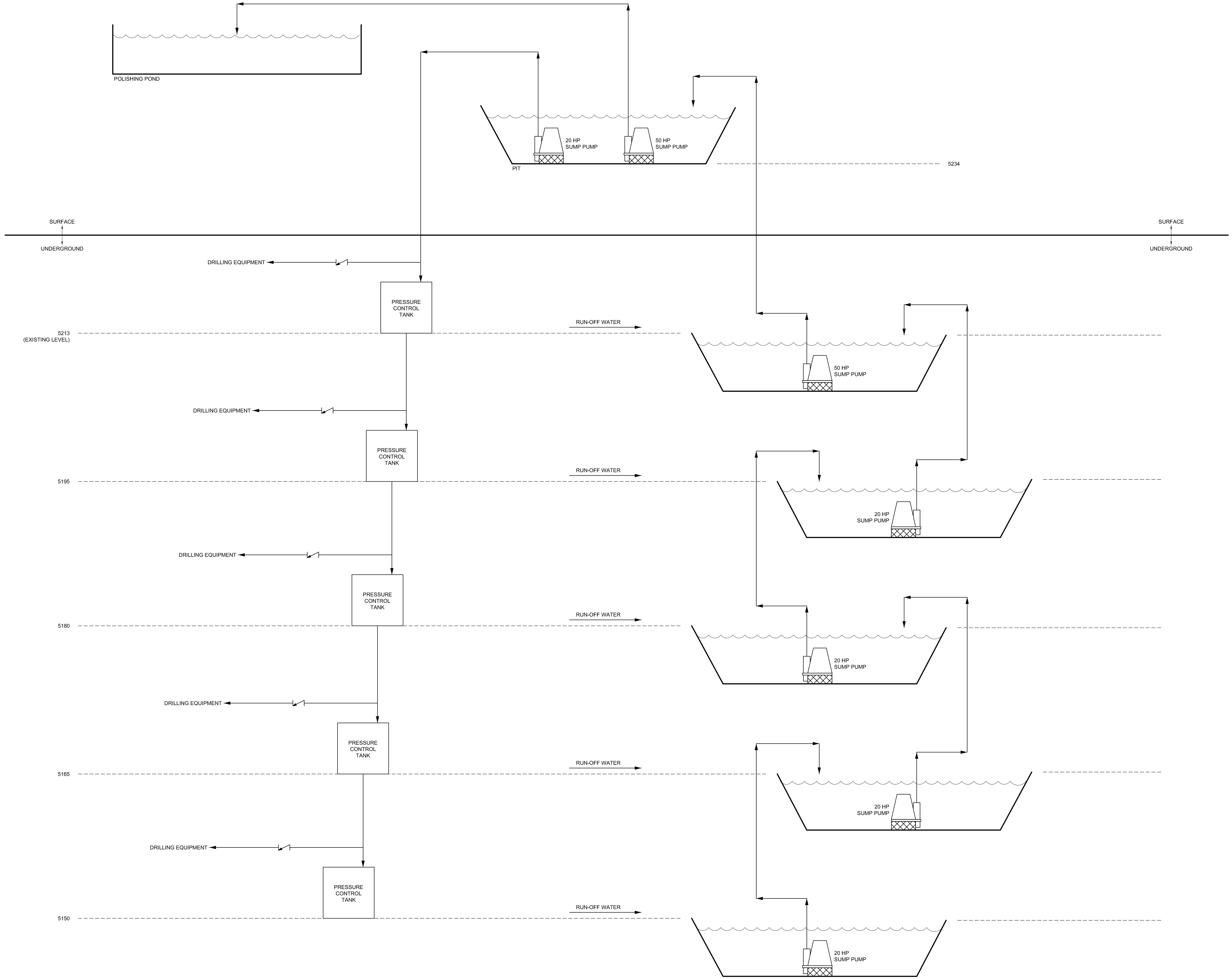

Client :

WALLBRIDGE MINING COMPANY LIMITED

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PRELIMINARY
DO NOT USE FOR CONSTRUCTION
SNC·LAVALIN 2017-01-11

REV.	DATE	ISSUES	BY	CHK	APP.
B00	2017-01-11	ISSUED FOR PREFEASIBILITY STUDY	B.LEM.	M.HÉN.	-
BPB	2016-12-14	ISSUED FOR FINAL REVIEW	B.LEM.	M.HÉN.	-
B	2016-11-11	ISSUED FOR COMMENTS	B.LEM.	M.HÉN.	-
A	2016-11-01	ISSUED FOR COMMENTS	B.LEM.	M.HÉN.	-

Seal(s) :

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150, rue Gamble Ouest
Rouyn-Noranda (Québec) J9X 2R7
Tel. : 819 764-5181
Fax : 819 797-0158
www.snclavalin.com

Project :

FENELON STUDY

Title :

SURFACE & UNDERGROUND
INFRASTRUCTURES
WATER DISTRIBUTION PUMP SYSTEM
-
-
FLOWSHEET

Drawn by : Bruno Lemieux, tech.	Org. Date : 2016-10-25
Designed by : Marc Hénault, P. eng.	Project manager : Marc Hénault, P. eng.
Verified by : Marc Hénault, P. eng.	Project # : 640914-0000
Approved by : Marc Hénault, P. eng.	Reference : -
Scale : N/A	Drawing # : 640914-0000-45DD-0200 — Rev. : B00

Tracé par: "Lemib", 2017-01-11 10:30:27 (Lts: 1.00 ; Mea: 1)

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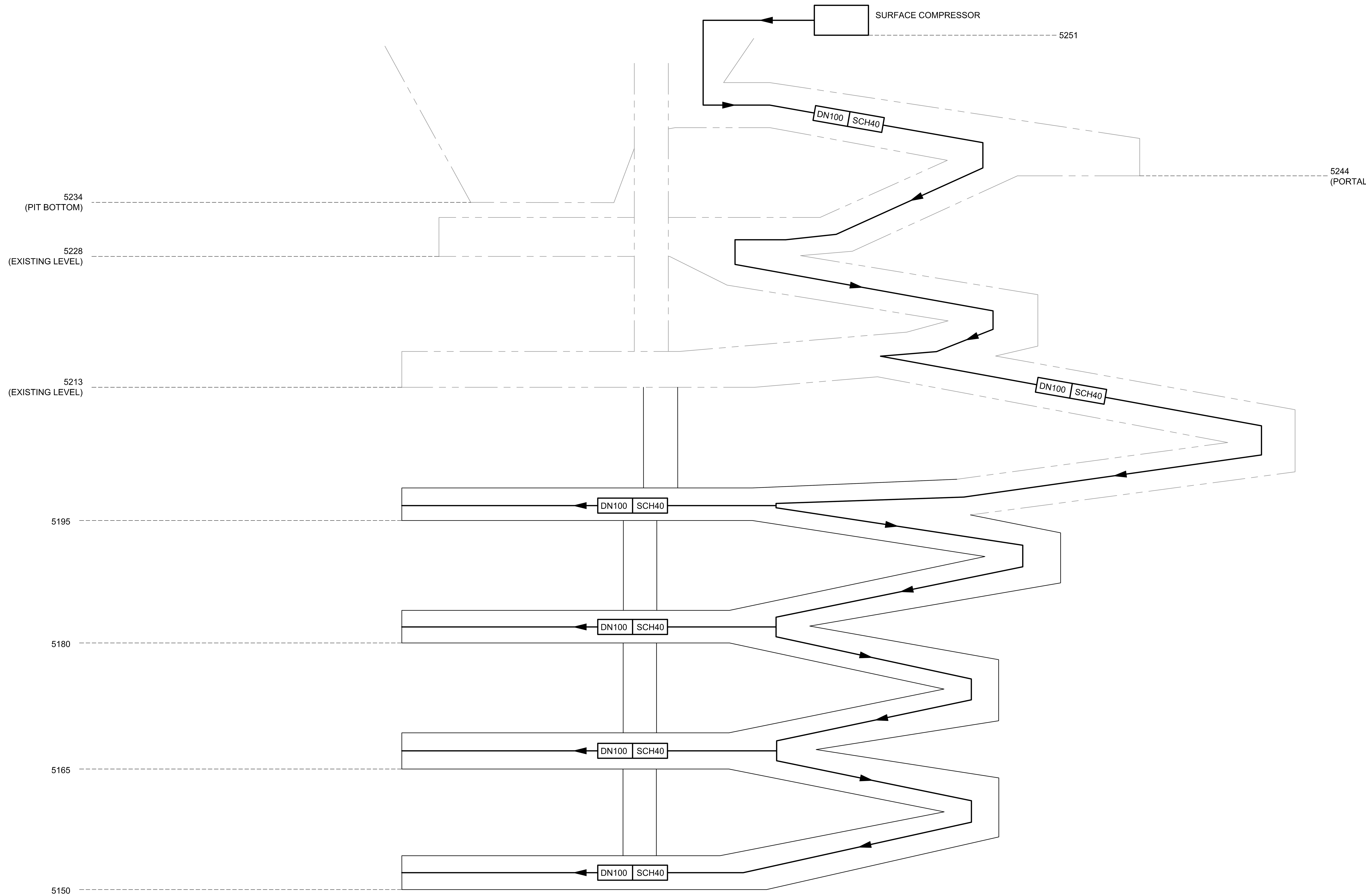

COMPRESSED AIR DISTRIBUTION SYSTEM
RAMP FLOW DIAGRAM

SCALEL : N/A

Client :

WALLBRIDGE MINING COMPANY LIMITED

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B00	2017-01-11	ISSUED FOR PREFEASIBILITY STUDY	B.LEM.	M.HÉN.	-
BPB	2016-12-14	ISSUED FOR FINAL REVIEW	B.LEM.	M.HÉN.	-
A	2016-11-11	ISSUED FOR COMMENTS	ST.MAR.	M.HÉN.	-

Seal(s) :

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Rouyn-Noranda (Québec) J9X 2R7
Tel. : 819 764-5181
Fax : 819 797-0158
www.snclavalin.com

Project :

FENELON STUDY

Title :

SURFACE & UNDERGROUND
INFRASTRUCTURES
COMPRESSED AIR DISTRIBUTION SYSTEM
-
-
FLOW DIAGRAM

Drawn by : Stéphane Marcotte, tech.	Org. Date : 2016-11-09
Designed by : Marc Hénault, P. eng.	Project manager : Marc Hénault, P. eng.
Verified by : Marc Hénault, P. eng.	Project # : 640914-0000
Approved by : Marc Hénault, P. eng.	Reference : -
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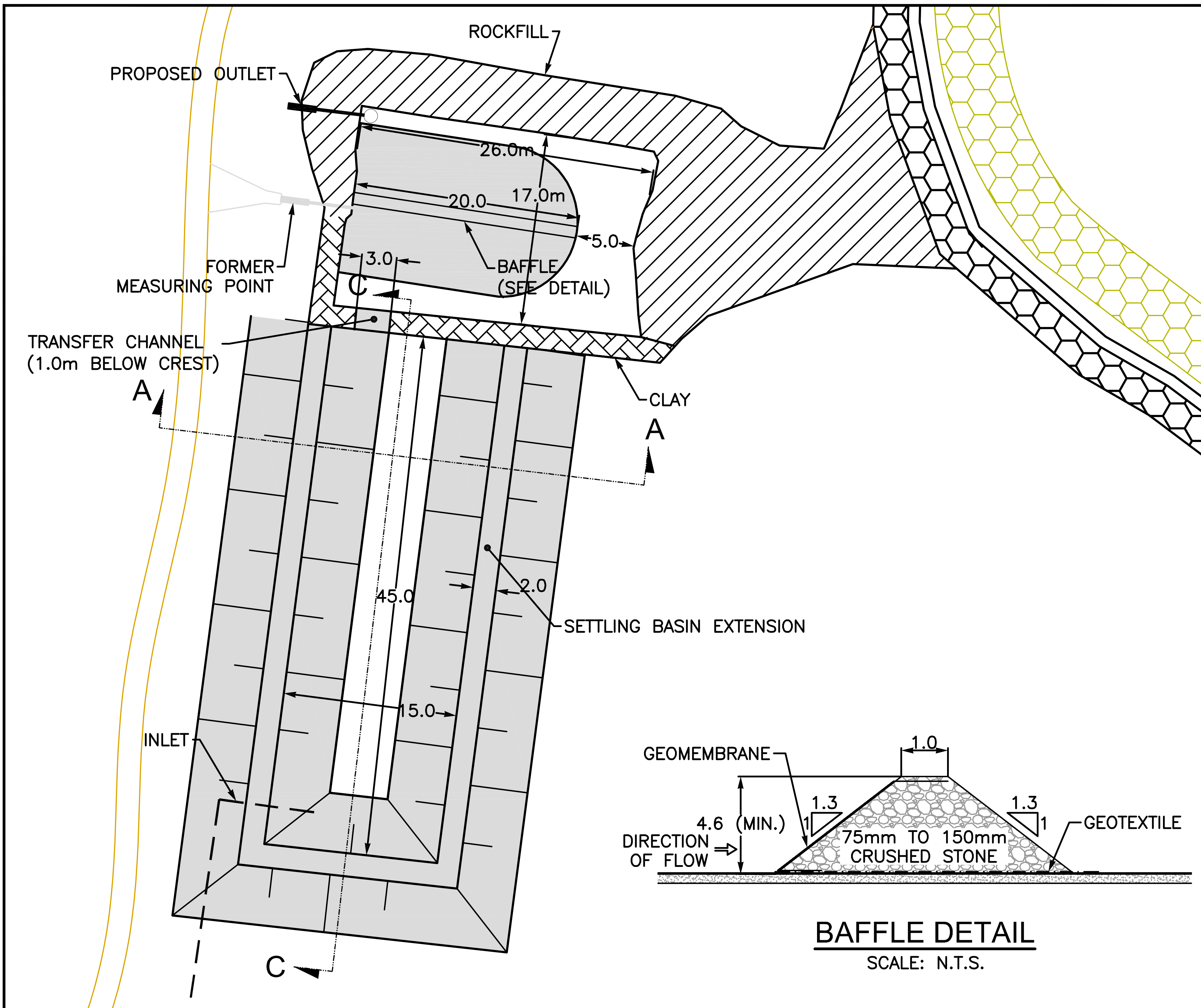
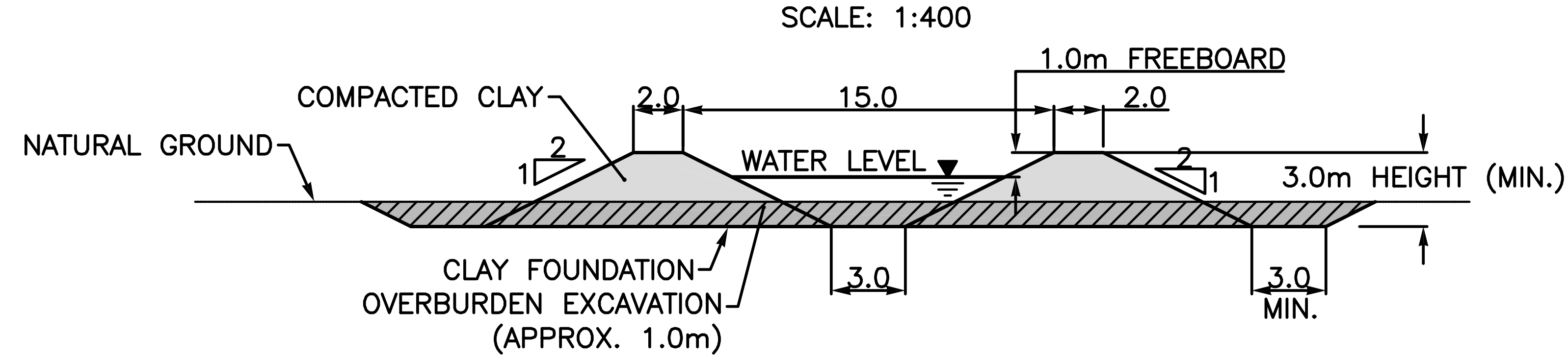


BAFFLE DETAIL

SCALE: N.T.S.

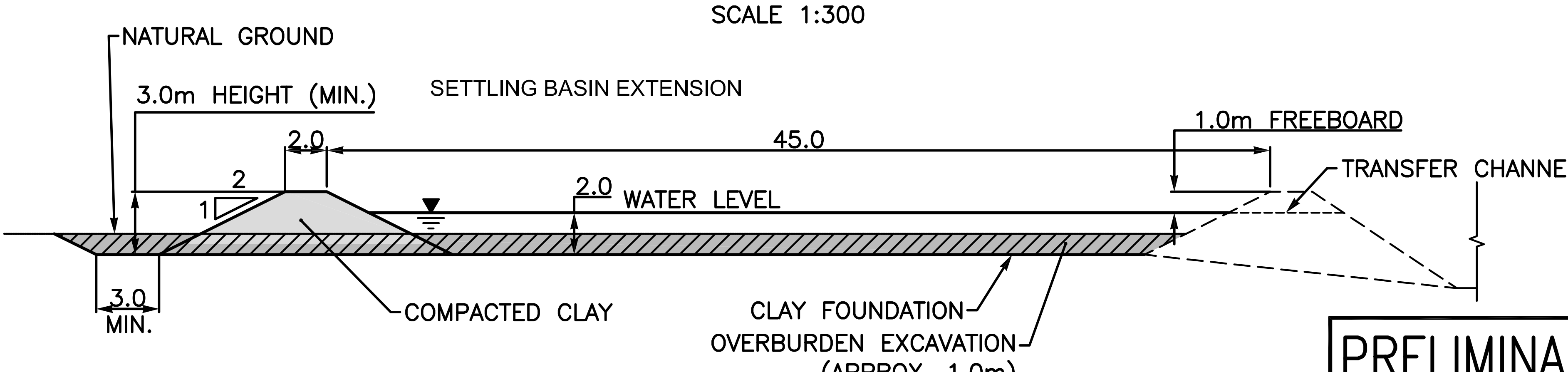
PLAN VIEW- SETTLING BASIN EXTENSION

SCALE: 1:400



SECTION A-A

SCALE 1:300



SECTION C-C

SCALE: 1:300

PRELIMINARY
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CLIENT:
WALLBRIDGE MINING COMPANY LIMITED

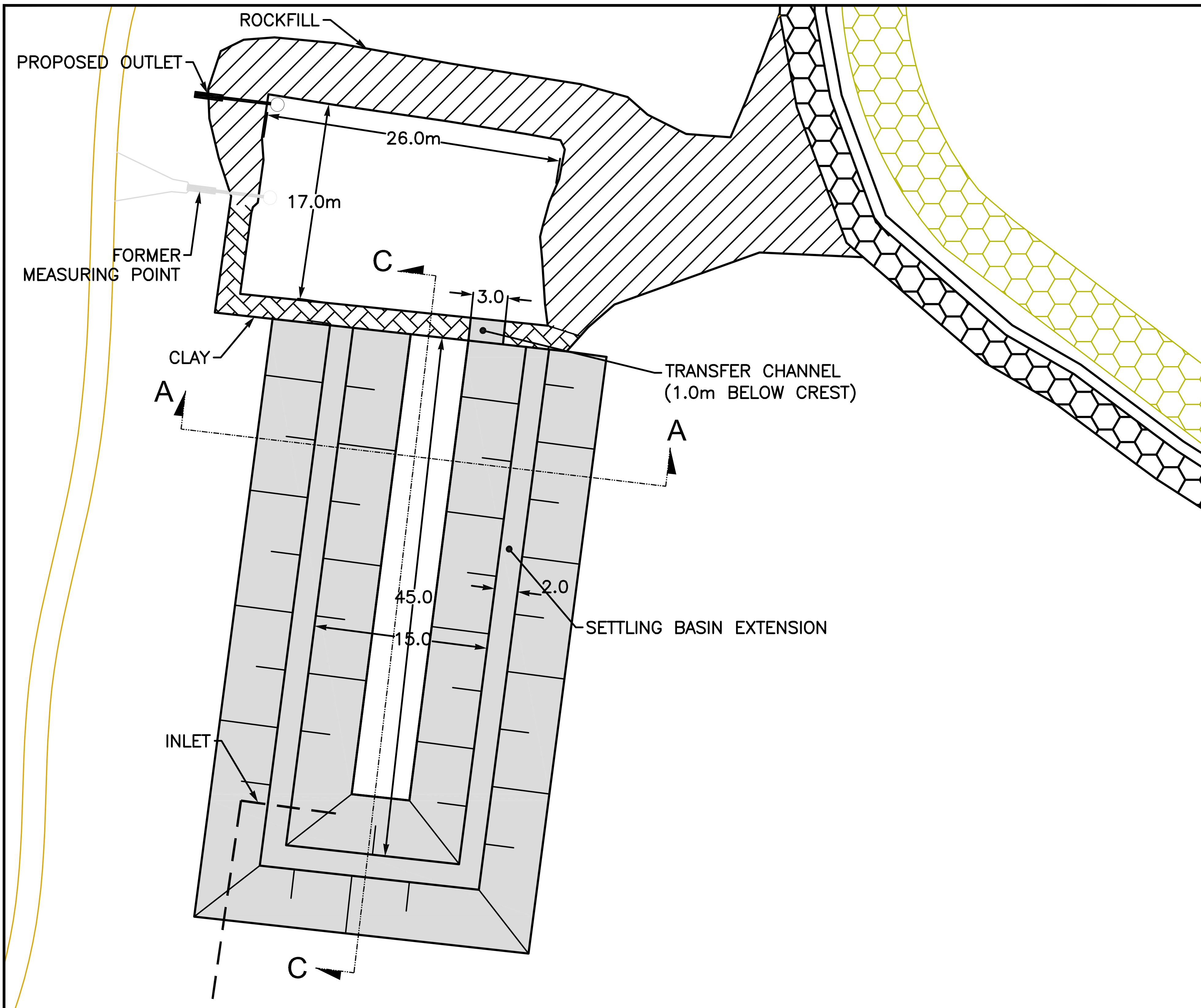
FENELON PROJECT
SETTLING BASIN EXTENSION
PLAN VIEW AND SECTIONS

SNC·LAVALIN
Développement Minier Durable
Mines et Métallurgie
SNC LAVALIN INC.
360, rue Saint-Jacques Ouest
Montréal (Québec)
Canada H2Y 1P5

DATE : 2017-01-12 ÉCHELLE : 1:250

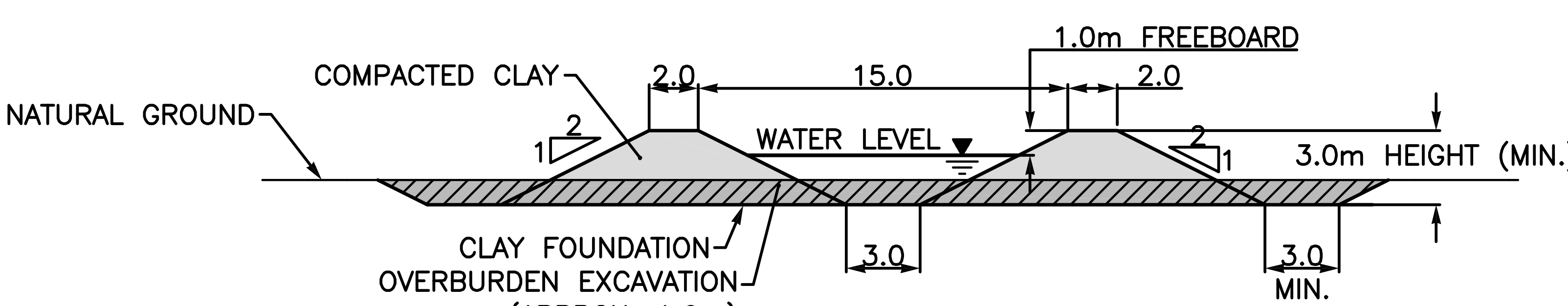
FAIT	VÉRIF.	APPR.	
D.L	M.M	J-LB	NO. FIGURE 1A

DERNIÈRE SAUVEGARDE: 2017/01/12 - 12:56pm
CHEMIN: P:\300-RN\640914\0000\0-Docs Emis\2017-01-12 - dessins pour Simon\Figure 1 - English -OPTION 1A.dwg



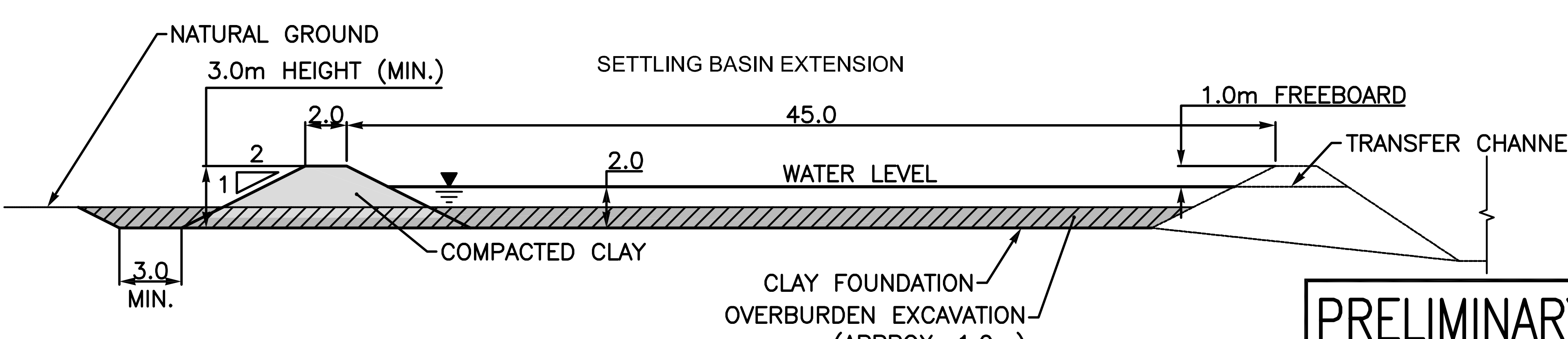
PLAN VIEW- SETTLING BASIN EXTENSION

SCALE: 1:400



SECTION A-A

SCALE 1:300



SECTION C-C

SCALE: 1:300

PRELIMINARY

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CLIENT:

WALLBRIDGE MINING COMPANY LIMITED

FENELON PROJECT
SETTLING BASIN EXTENSION
PLAN VIEW AND SECTIONS

SNC·LAVALIN

Développement Minier Durable Mines et Metallurgie
SNC LAVALIN INC.
360, rue Saint-Jacques Ouest
Montréal (Québec)
Canada H2Y 1P5

DATE : 2017-01-12	ÉCHELLE : 1:250

FAIT	VÉRIF.	APPR.	NO. FIGURE 1B
D.L.	M.M	J-L.B	

DERNIÈRE SAUVEGARDE: 2017/01/12 – 12:56pm
CHEMIN: P:\300-RN\640914\0000\0-Docs Emis\2017-01-12 – dessins pour Simon\Figure 1 – OPTION 1B.dwg

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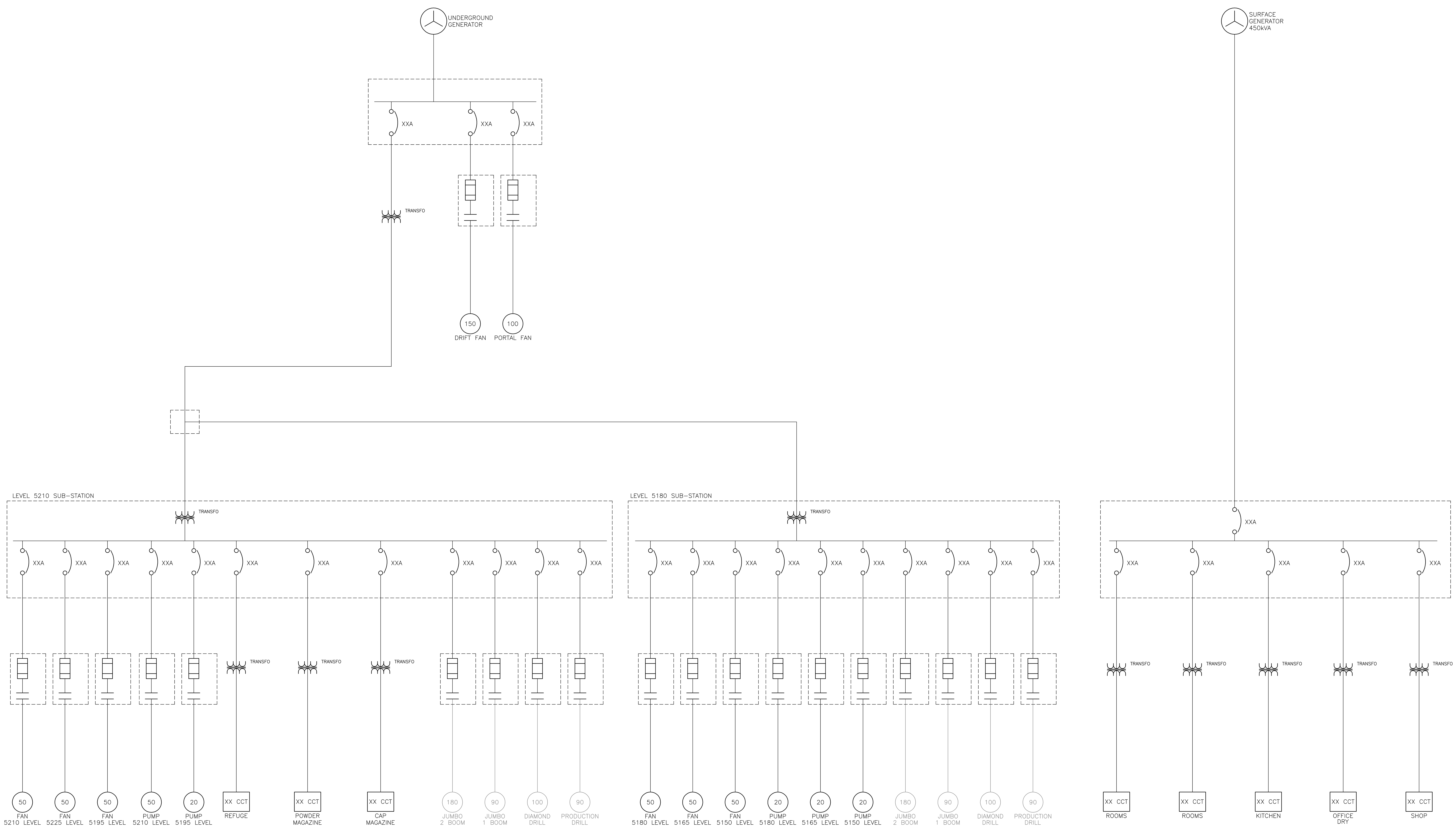

UNDERGROUND SINGLE LINE DIAGRAM

ÉCHELLE : N/A

SURFACE SINGLE LINE DIAGRAM

ÉCHELLE : N/A

Client :

WALLBRIDGE MINING COMPANY LIMITED

Key Plan :

Professionals :

POUR INFORMATION
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SNC·LAVALIN 2016-11-15

REV.	DATE	ISSUES	BY	CHK	APP.
0	15 NOV 2016	ISSUED FOR PRE FEASIBILITY STUDY	P.CHA	P.FOR	P.FOR

Seal(s) :

SNC·LAVALIN

SNC-Lavalin Stavibel inc.
150, rue Gamble Ouest
Rouyn-Noranda (Québec) J9X 2R7
Tel. : 819 764-5181
Fax : 819 797-0158
www.snclavalin.com

Project :

FENELON STUDY

Title :

UNDERGROUND INFRASTRUCTURES
-
SURFACE & UNDERGROUND DISTRIBUTION SINGLE LINE DIAGRAM

Drawn by : PIERRE CHAMBERLAND	Org. Date : 2016-11-10
Designed by : PIERRE CHAMBERLAND	Project manager : PIERRE FORTIN, ING.
Verified by : PIERRE FORTIN, ING.	Project # : 640914-0000
Approved by : PIERRE FORTIN, ING.	Reference : -
Scale : N/A	Drawing # : 640914-0000-47D1-0001 — Rev. : 0

Tracé par: "Champ3", --- (Lts: 0.35 ; Mea: 1)

SNC·LAVALIN

Equipments												Consumed Load						
				Nameplate			Load data					kW= Absorbed Load / Efficiency			kVAR=kW*tan(acos(PF))			
Equipment Number	Description	Charge type (**N**ormal or **U**rgency)	Operation mode (**C**ontinous, **I**ntermittent or **S**tandby)	HP	kVA	kW	Load factor	Efficiency	Power Factor	Load in kW	Absorbed Power kW	Continous		Intermittent		Standby		Remarks
												kW	kVAR	kW	kVAR	kW	kVAR	
	OFFICE/DRY	N	C			50,0	85,00	100,00	0,95	50,00	4250,00	42,50	13,97					
	KITCHEN	N	C			50,0	85,00	100,00	0,95	50,00	4250,00	42,50	13,97					
	ROOMS	N	C			40,0	85,00	100,00	0,95	40,00	3400,00	34,00	11,18					
	ROOMS	N	C			40,0	85,00	100,00	0,95	40,00	3400,00	34,00	11,18					
	SHOP	N	C			50,0	85,00	100,00	0,95	50,00	4250,00	42,50	13,97					

Operation factor X = 100 %
Y = 75 %
Z = 10 %

			FP
Operation Load total (OLT) :			
kW = X% * Total kW C + Y% * Total kW I	**196 kW**	**206 kVA**	**95,00%**
kVAR = X% * Total kVAR C + Y% * Total kVAR I	**64 kVAR**		
Peek load total (PLT) :			
kW = X% * Total kW C + Y% * Total kW I + Z% * Total kW R	**196 kW**	**206 kVA**	**95,00%**
kVAR = X% * Total kVAR C + Y% * Total kVAR I + Z% * Total kVAR R	**64 kVAR**		
Conception Load (CL) :			
kW = 110% * kW COT + Z% * Total kW R	**215 kW**	**226 kVA**	**95,00%**
kVAR = 110% * COT kVAR + Z% * Total kVAR R	**71 kVAR**		

	Continue		***Intermittante***		***De réserve***	
Total	195,50	64,26	0,00	0,00	0,00	0,00
Total kVA	206		0		0	
Total Normal	195,50	64,26	0,00	0,00	0,00	0,00
Normal kVA	206		0		0	
Total Urgency	0,00	0,00	0,00	0,00	0,00	0,00
Urgency kVA	0		0		0	

REFERENCE:	REV.	DESCRIPTION	DATE	PAR	VÉRIFIÉ PAR	APPR. PAR	CLIENT
	0	Pre feasibility study	15-11-16	P.C	P.F	P.F	

Fenelon Gold Project
Building Load List
640914-0000-47EL-0001
Sheet 1/4

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Equipments												Consumed Load						SNC·LAVALIN
				Nameplate			Load data					kW= Absorbed Load / Efficiency			kVAR=kW*tan(acos(PF))			
Equipment Number	Description	Charge type (**N**ormal or **U**rgency)	Operation mode (**C**ontinous, **I**ntermittent or **S**tandby)	HP	kVA	kW	Load factor	Efficiency	Power Factor	Load in kW	Absorbed Power kW	Continous		Intermittent		Standby		Remarks
												kW	kVAR	kW	kVAR	kW	kVAR	
	5210 Sub-Station	N	C		431,0		100,00	100,00	0,82	355,14	35514,40	355,14	244,20					
	5280 Sub-Station	N	C		132,0		100,00	100,00	0,80	105,60	10560,00	105,60	79,20					
	DRIFT FAN	N	C	150,0			85,00	95,00	0,80	111,90	9511,50	100,12	75,09					
	PORTAL FAN	N	C	100,0			85,00	95,00	0,80	74,60	6341,00	66,75	50,06					

Operation factor X = 100 %, Y = 75 %, Z = 10 %

			FP
Operation Load total (OLT) :			
kW = X% * Total kW C + Y% * Total kW I	**628 kW**	**771 kVA**	**81,36%**
kVAR = X% * Total kVAR C + Y% * Total kVAR I	**449 kVAR**		
Peek load total (PLT) :			
kW = X% * Total kW C + Y% * Total kW I + Z% * Total kW R	**628 kW**	**771 kVA**	**81,36%**
kVAR = X% * Total kVAR C + Y% * Total kVAR I + Z% * Total kVAR R	**449 kVAR**		
Conception Load (CL) :			
kW = 110% * kW COT + Z% * Total kW R	**690 kW**	**849 kVA**	**81,36%**
kVAR = 110% * COT kVAR + Z% * Total kVAR R	**493 kVAR**		

	Continue		***Intermittante***		***De réserve***	
Total	627,61	448,55	0,00	0,00	0,00	0,00
Total kVA	771		0		0	
Total Normal	627,61	448,55	0,00	0,00	0,00	0,00
Normal kVA	771		0		0	
Total Urgency	0,00	0,00	0,00	0,00	0,00	0,00
Urgency kVA	0		0		0	

REFERENCE:	REV.	DESCRIPTION	DATE	PAR	VÉRIFIÉ PAR	APPR. PAR	CLIENT
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Fenelon Gold Project
Underground Load List
640914-0000-47EL-0001
Sheet 2/4

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Equipments												Consumed Load						
				Nameplate			Load data					kW= Absorbed Load / Efficiency			kVAR=kW*tan(acos(PF))			SNC·LAVALIN
Equipment Number	Description	Charge type (**N**ormal or **U**rgency)	Operation mode (**C**ontinous, **I**ntermittent or **S**tandby)	HP	kVA	kW	Load factor	Efficiency	Power Factor	Load in kW	Absorbed Power kW	Continous		Intermittent		Standby		Remarks
												kW	kVAR	kW	kVAR	kW	kVAR	
	FAN - LEVEL 5210	N	I	50,0			85,00	95,00	0,80	37,30	3170,50			33,37	25,03			
	FAN - LEVEL 5225	N	I	50,0			85,00	95,00	0,80	37,30	3170,50			33,37	25,03			
	FAN - LEVEL 5195	N	I	50,0			85,00	95,00	0,80	37,30	3170,50			33,37	25,03			
	PUMP - LEVEL 5210	N	I	50,0			85,00	95,00	0,80	37,30	3170,50			33,37	25,03			
	PUMP - LEVEL 5195	N	I	20,0			85,00	93,00	0,80	14,92	1268,20			13,64	10,23			
	JUMBO 2 BOOM	N	I	180,0			85,00	95,00	0,80	134,28	11413,80			120,15	90,11			
	JUMBO 1 BOOM	N	I	90,0			85,00	95,00	0,80	67,14	5706,90			60,07	45,05			
	DIAMOND DRILL	N	I	100,0			85,00	95,00	0,80	74,60	6341,00			66,75	50,06			
	PRODUCTION DRILL	N	I	90,0			85,00	95,00	0,80	67,14	5706,90			60,07	45,05			
	POWDER MAGAZINE	N	C			0,5	85,00	100,00	0,95	0,50	42,50	0,43	0,14					
	CAP MAGAZINE	N	C			0,5	85,00	100,00	0,95	0,50	42,50	0,43	0,14					
	REFUGE	N	C			5,0	85,00	100,00	0,95	5,00	425,00	4,25	1,40					

Operation factor	X = 100 % Y = 75 % Z = 10 %		
Operation Load total (OLT) :			**FP**
kW = X% * Total kW C + Y% * Total kW I	**346 kW**	**431 kVA**	**80,24%**
kVAR = X% * Total kVAR C + Y% * Total kVAR I	**257 kVAR**		
Peek load total (PLT) :			
kW = X% * Total kW C + Y% * Total kW I + Z% * Total kW R	**346 kW**	**431 kVA**	**80,24%**
kVAR = X% * Total kVAR C + Y% * Total kVAR I + Z% * Total kVAR R	**257 kVAR**		
Conception Load (CL) :			
kW = 110% * kW COT + Z% * Total kW R	**380 kW**	**474 kVA**	**80,24%**
kVAR = 110% * COT kVAR + Z% * Total kVAR R	**283 kVAR**		

	Continue		*Intermittante*		*De réserve*	
Total	5,10	1,68	454,17	340,63	0,00	0,00
Total kVA	5		568		0	
Total Normal	5,10	1,68	454,17	340,63	0,00	0,00
Normal kVA	5		568		0	
Total Urgency	0,00	0,00	0,00	0,00	0,00	0,00
Urgency kVA	0		0		0	

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Fenelon Gold Project
Underground 5210 Load List
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Sheet 3/4

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Equipments												Consumed Load						SNC·LAVALIN
				Nameplate			Load data					kW= Absorbed Load / Efficiency			kVAR=kW*tan(acos(PF))			
Equipment Number	Description	Charge type (**N**ormal or **U**rgency)	Operation mode (**C**ontinous, **I**ntermittent or **S**tandby)	HP	kVA	kW	Load factor	Efficiency	Power Factor	Load in kW	Absorbed Power kW	Continous		Intermittent		Standby		Remarks
												kW	kVAR	kW	kVAR	kW	kVAR	
	FAN - LEVEL 5180	N	I	50,0			85,00	95,00	0,80	37,30	3170,50			33,37	25,03			
	FAN - LEVEL 5165	N	I	50,0			85,00	95,00	0,80	37,30	3170,50			33,37	25,03			
	FAN - LEVEL 5210	N	I	50,0			85,00	95,00	0,80	37,30	3170,50			33,37	25,03			
	PUMP - LEVEL 5180	N	I	20,0			85,00	93,00	0,80	14,92	1268,20			13,64	10,23			
	PUMP - LEVEL 5165	N	I	20,0			85,00	93,00	0,80	14,92	1268,20			13,64	10,23			
	PUMP - LEVEL 5150	N	I	20,0			85,00	93,00	0,80	14,92	1268,20			13,64	10,23			
	JUMBO 2 BOOM	N		180,0			85,00	95,00	0,80	134,28	11413,80							Only recorded on 1 level
	JUMBO 1 BOOM	N		90,0			85,00	95,00	0,80	67,14	5706,90							Only recorded on 1 level
	DIAMOND DRILL	N		100,0			85,00	95,00	0,80	74,60	6341,00							Only recorded on 1 level
	PRODUCTION DRILL	N		90,0			85,00	95,00	0,80	67,14	5706,90							Only recorded on 1 level

Operation factor X = 100 %
Y = 75 %
Z = 10 %

Operation Load total (OLT) :			***FP***
kW = X% * Total kW C + Y% * Total kW I	**106 kW**	**132 kVA**	**80,00%**
kVAR = X% * Total kVAR C + Y% * Total kVAR I	**79 kVAR**		
Peek load total (PLT) :			
kW = X% * Total kW C + Y% * Total kW I + Z% * Total kW R	**106 kW**	**132 kVA**	**80,00%**
kVAR = X% * Total kVAR C + Y% * Total kVAR I + Z% * Total kVAR R	**79 kVAR**		
Conception Load (CL) :			
kW = 110% * kW COT + Z% * Total kW R	**116 kW**	**145 kVA**	**80,00%**
kVAR = 110% * COT kVAR + Z% * Total kVAR R	**87 kVAR**		

	Continue		***Intermittent***		***Standby***	
Total	0,00	0,00	141,03	105,77	0,00	0,00
Total kVA	0		176		0	
Total Normal	0,00	0,00	141,03	105,77	0,00	0,00
Normal kVA	0		176		0	
Total Urgency	0,00	0,00	0,00	0,00	0,00	0,00
Urgency kVA	0		0		0	

	0	Pre feasibility study	15-11-16	P.C	P.F	P.F	
REFERENCE:	REV.	DESCRIPTION	DATE	PAR	VÉRIFIÉ PAR	APPR. PAR	CLIENT

Fenelon Gold Project
Underground 5180 Load List
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Sheet 4/4

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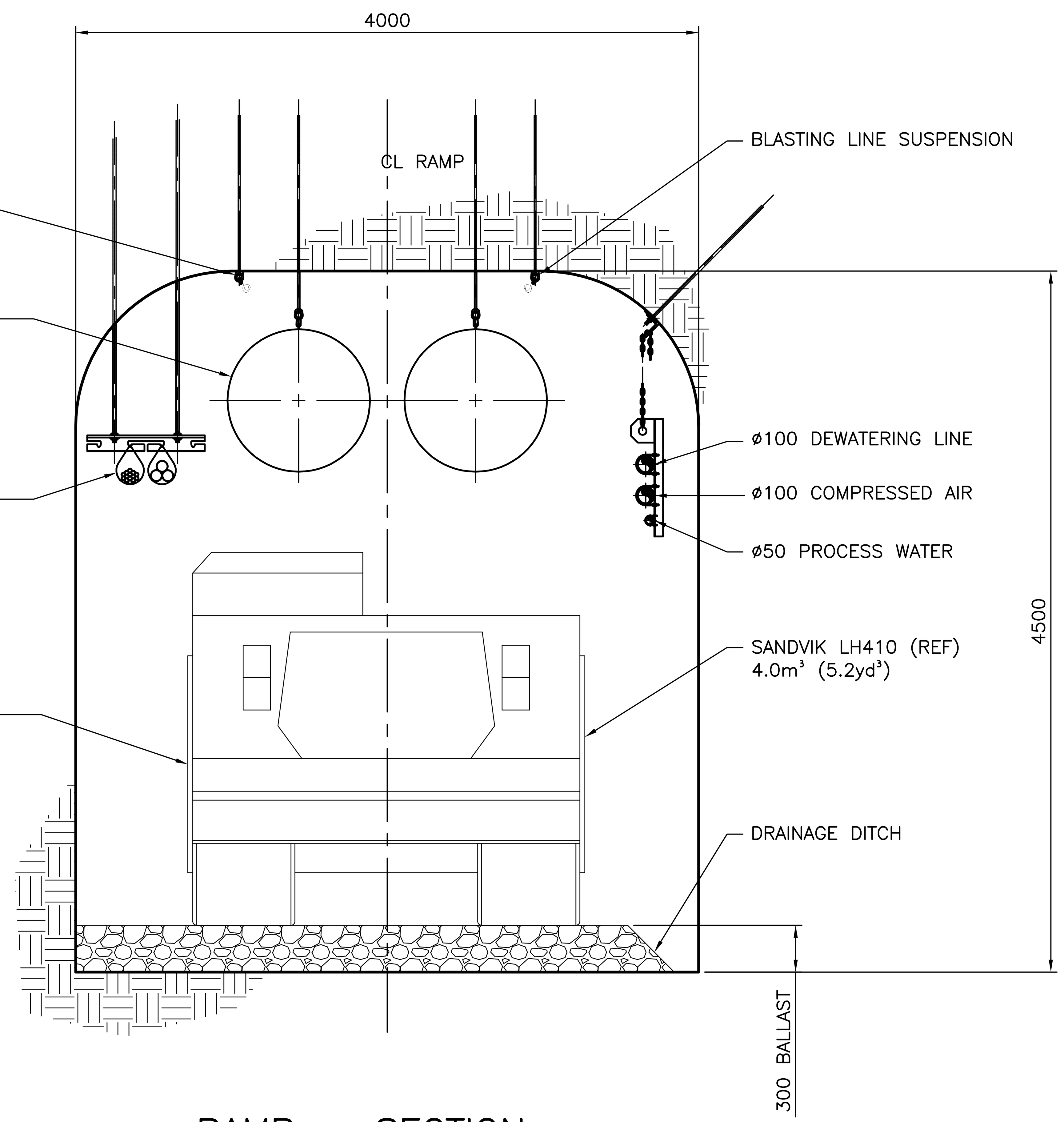

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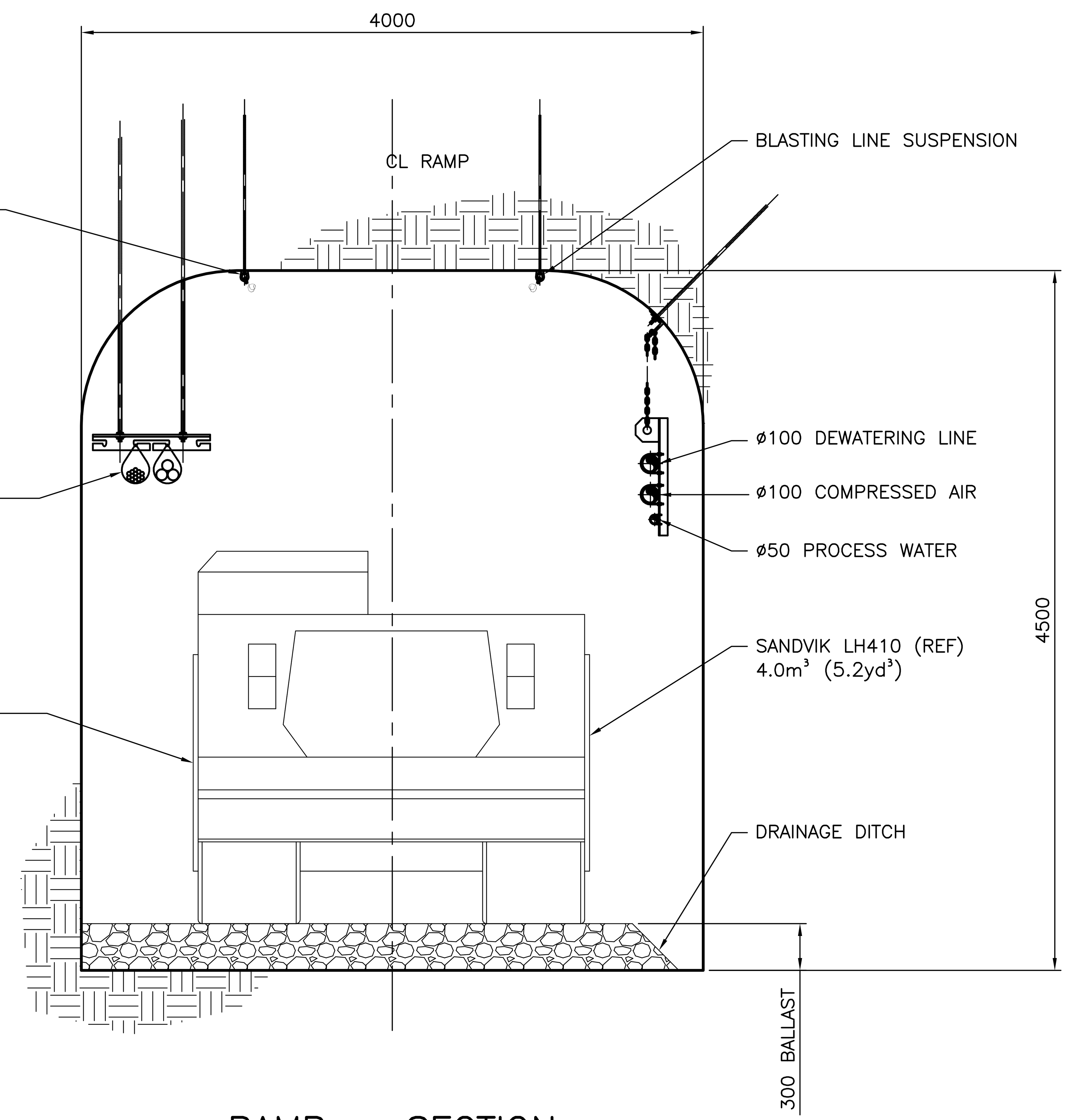

RAMP – SECTION

PRODUCTION PHASE

ISSUE No	REV.	DATE (Y/M/D)	PURPOSE OF ISSUE	TRANSMISSION LETTER No

ISSUE REGISTER

ISSUE No	REV.	DATE (Y/M/D)	PURPOSE OF ISSUE	TRANSMISSION LETTER No

ISSUE REGISTER

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B00	ISSUED FOR PRE-FEASIBILITY	17/01/12		
BPB	ISSUED FOR CLIENT REVIEW	16/12/14	RFL	
BPA	ISSUED FOR COMMENTS	16/11/04	RFL	

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40 Larch Street, Suite 300
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DATE 16/10/31	CLIENT
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CLIENT WALLBRIDGE MINING COMPANY LIMITED

PROJECT FENELON PRE–FEASIBILITY STUDY

TITLE UNDERGROUND INFRASTRUCTURE
DEVELOPMENT & PRODUCTION PHASES
RAMP
SECTIONS

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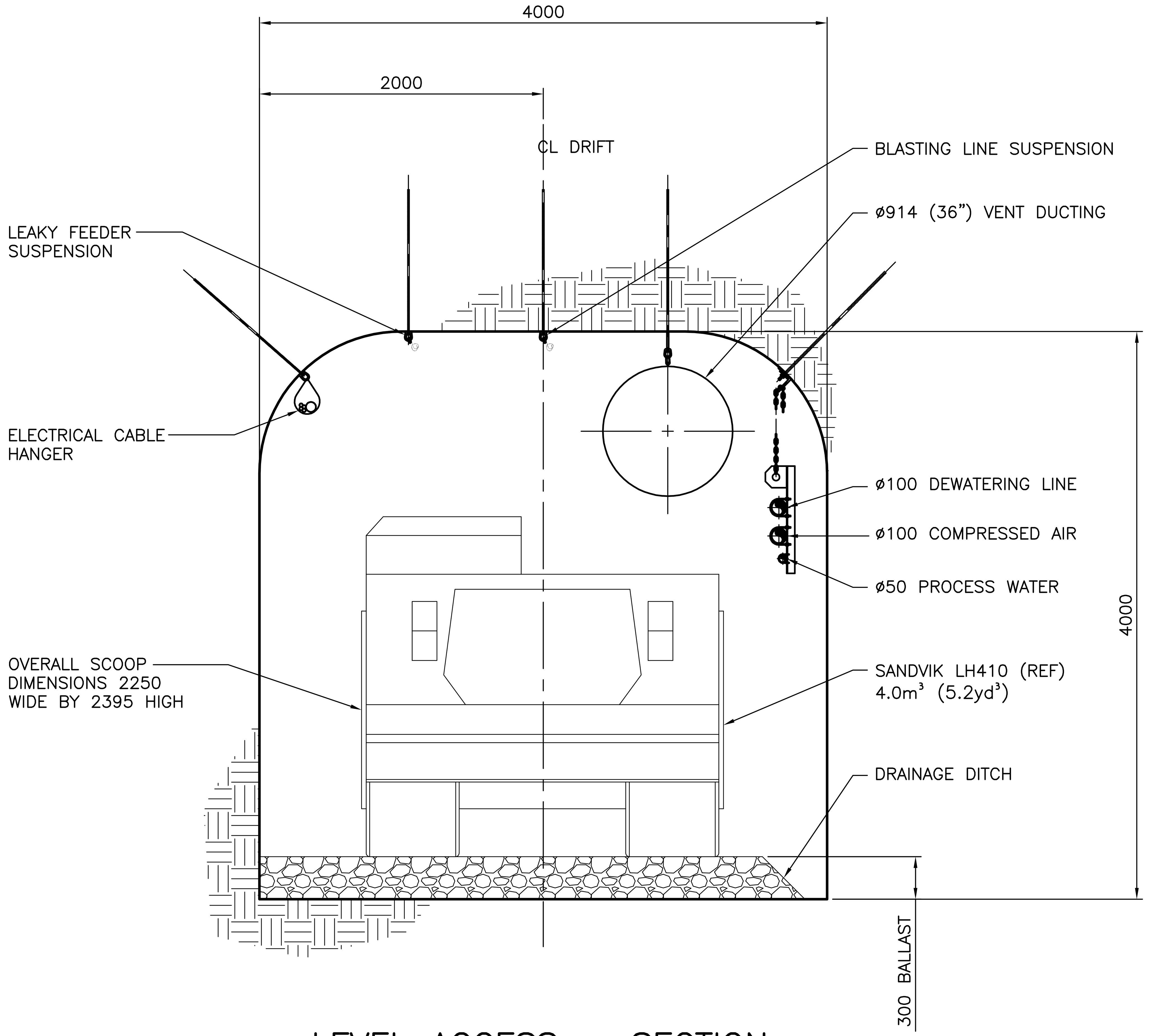

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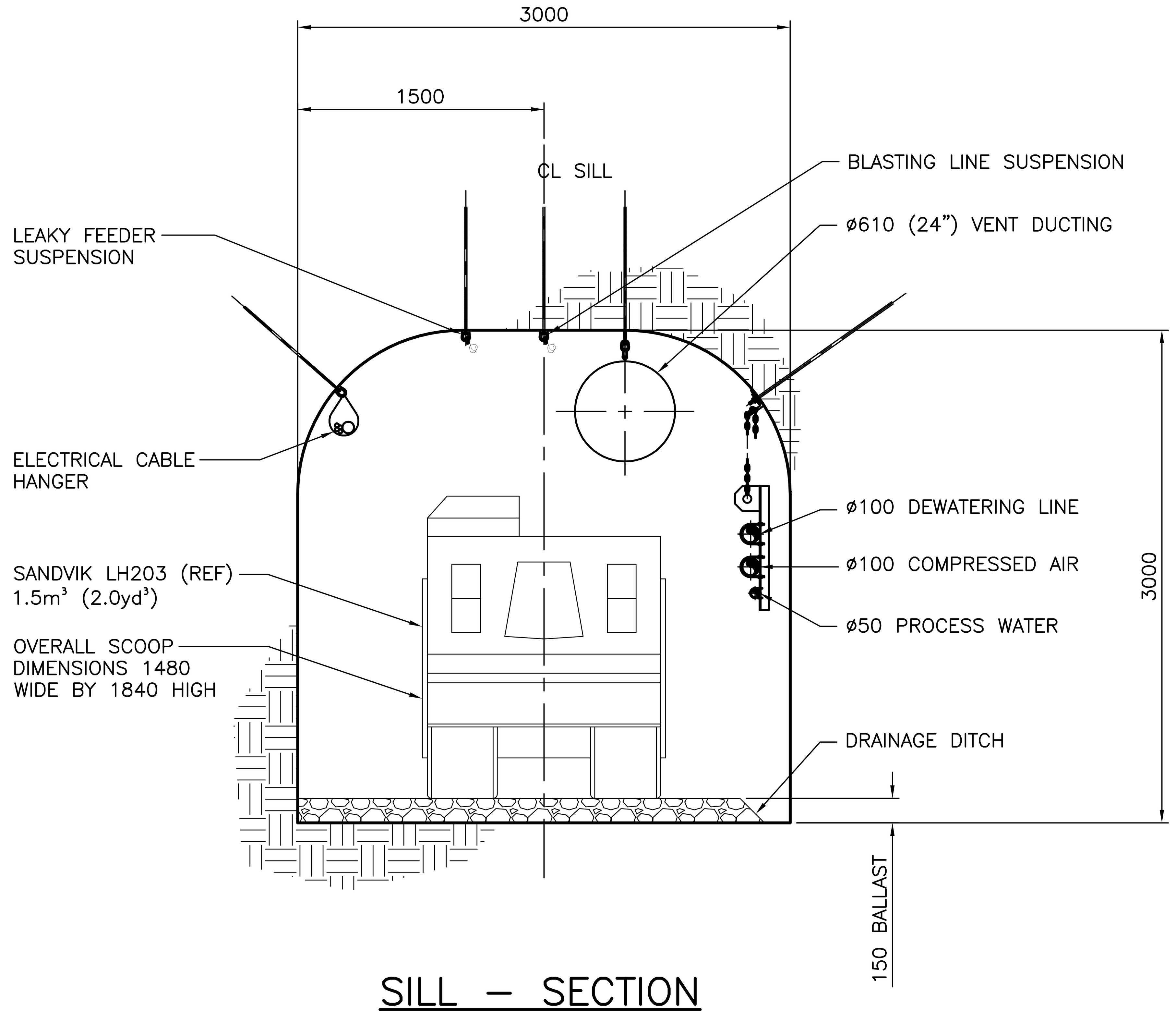

SILL – SECTION

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PROJECT
FENELON PRE-FEASIBILITY STUDY

TITLE
UNDERGROUND INFRASTRUCTURE
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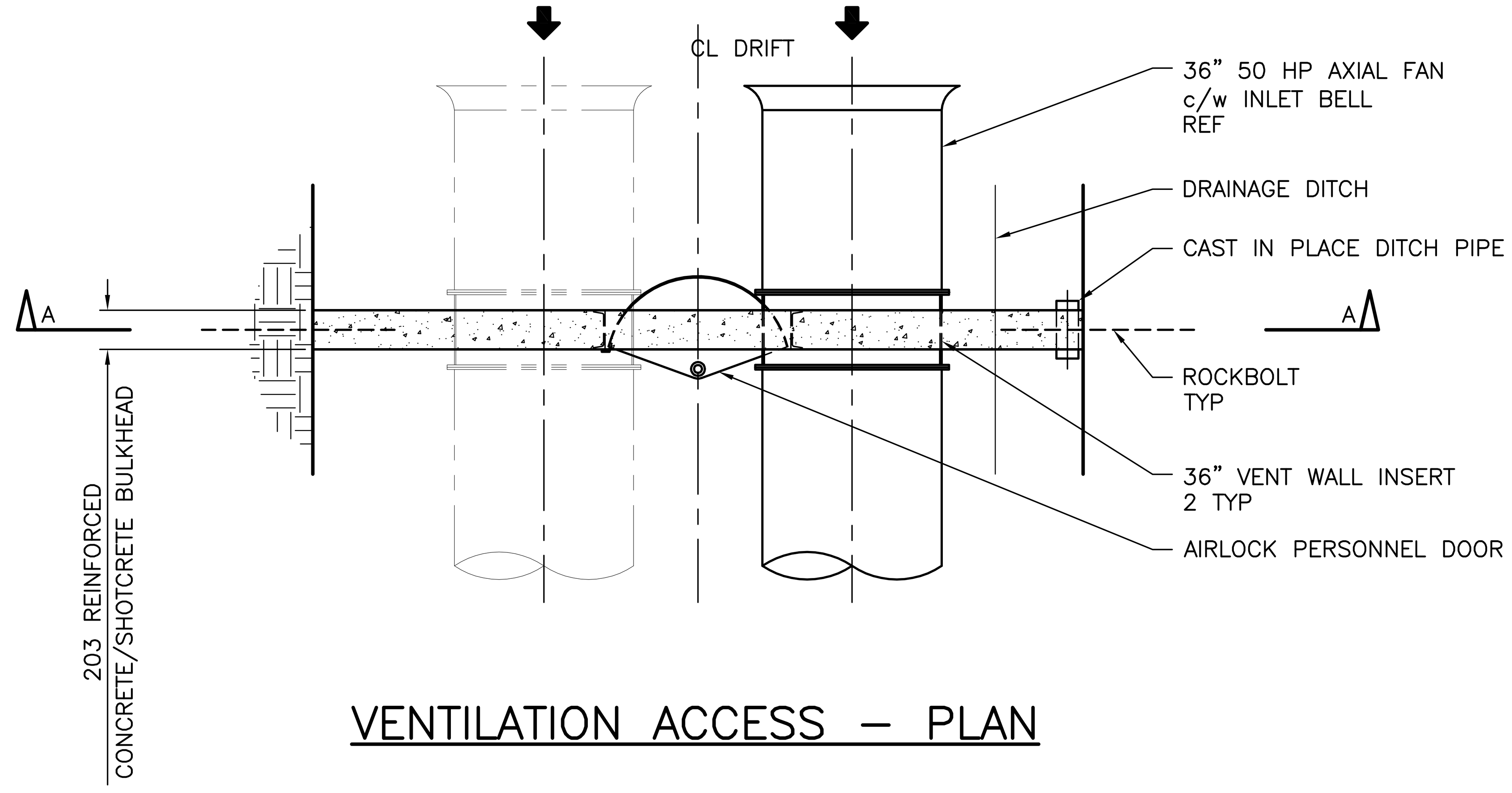
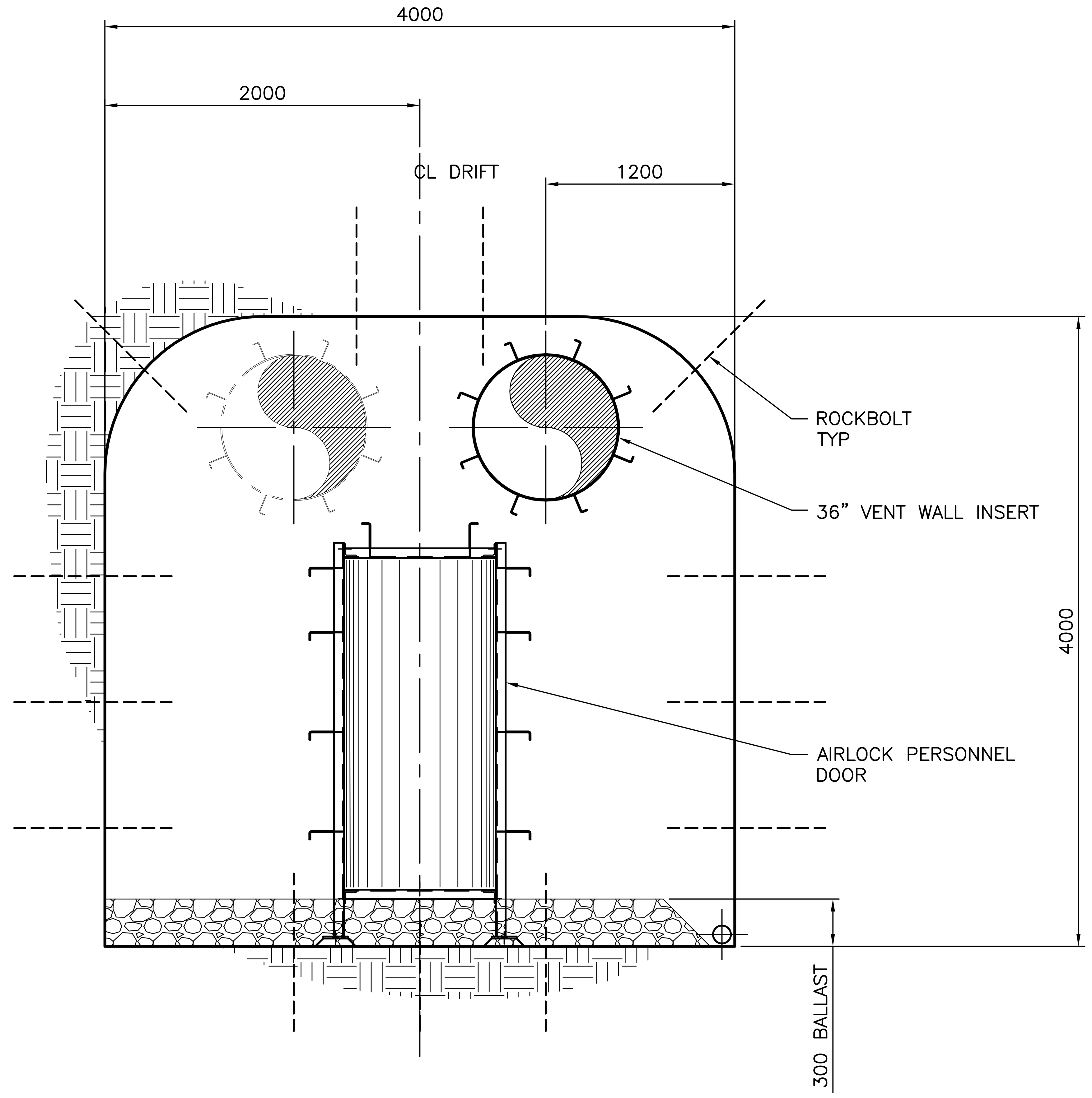


VENTILATION ACCESS – PLAN



SECTION AA

WALLBRIDGE MINING COMPANY LIMITED

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FENELON PRE-FEASIBILITY STUDY

TITLE
UNDERGROUND INFRASTRUCTURE
VENTILATION BULKHEAD
TYPICAL
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PROJECT No	SUBDIVISION	SUBJECT	SERIAL	REV.
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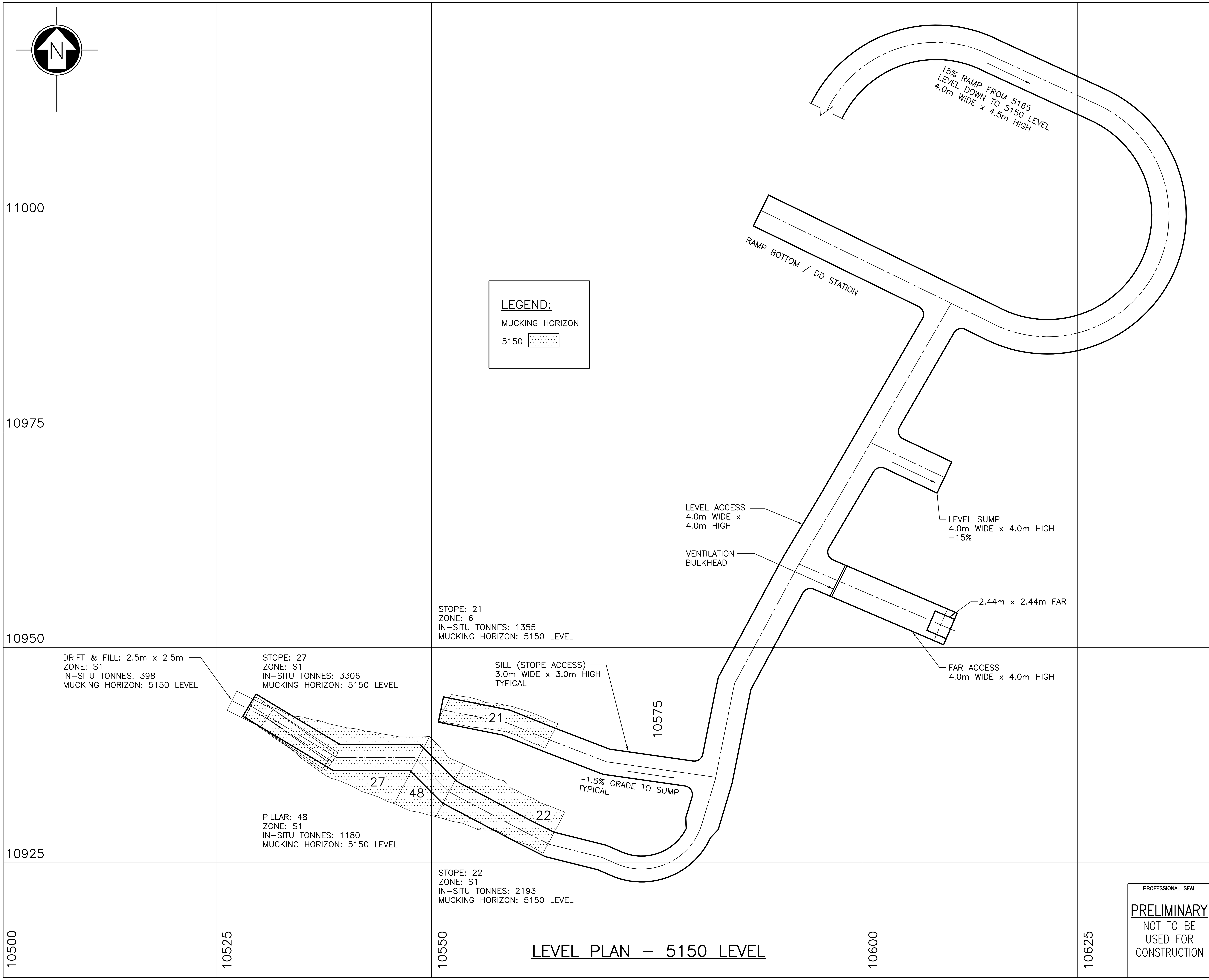

LEVEL PLAN – 5150 LEVEL

WALLBRIDGE MINING COMPANY LIMITED

ISSUE REGISTER

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PROJECT
FENELON GOLD MINE

TITLE
UNDERGROUND MINE DESIGN
TYPICAL LAYOUT
5150 LEVEL
PLAN

PROJECT No	SUBDIVISION	SUBJECT	SERIAL	REV.
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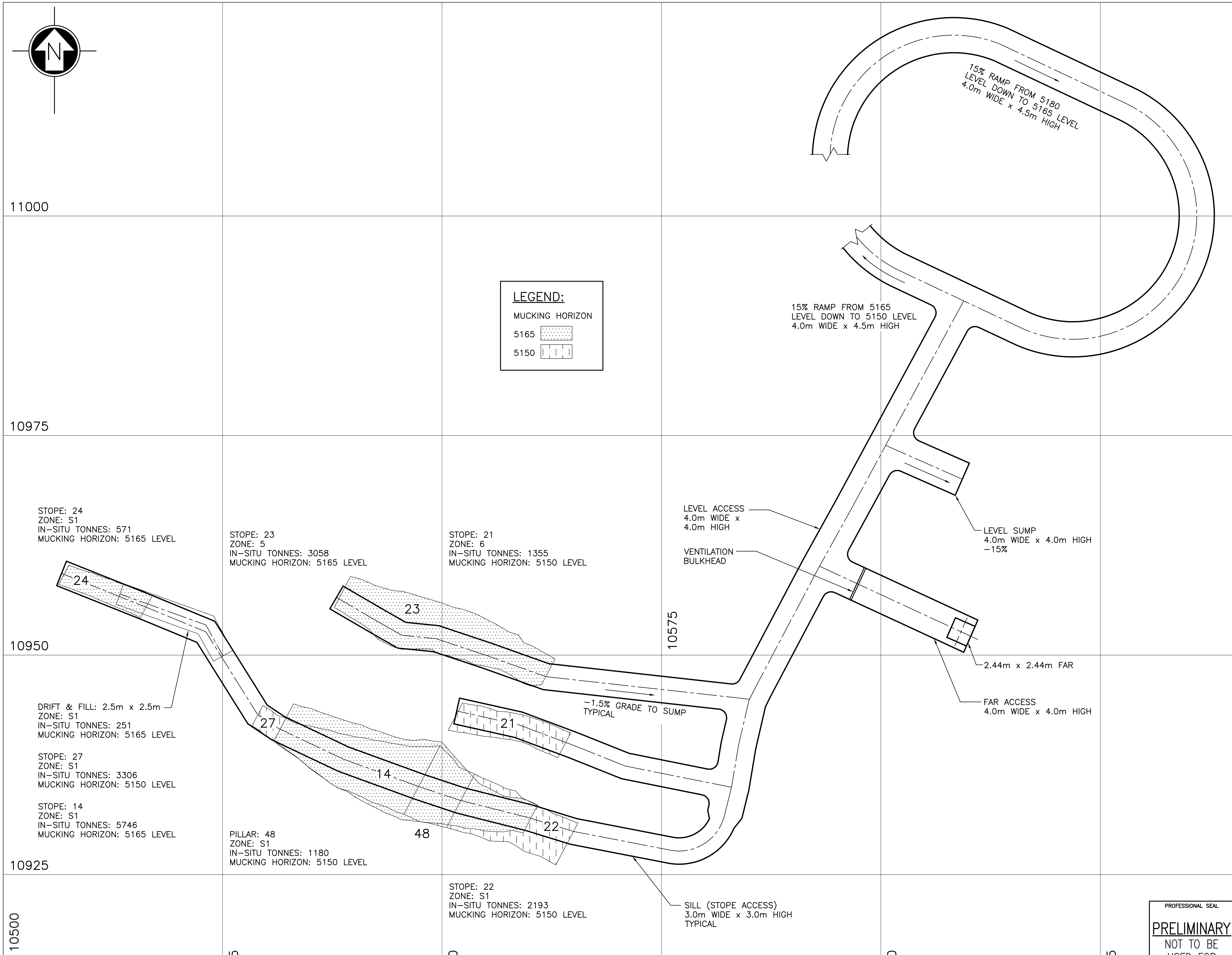

LEVEL PLAN – 5165 LEVEL

WALLBRIDGE MINING COMPANY LIMITED

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PROJECT
FENELON GOLD MINE

TITLE
UNDERGROUND MINE DESIGN
TYPICAL LAYOUT
5165 LEVEL
PLAN

PROJECT No	SUBDIVISION	SUBJECT	SERIAL	REV.
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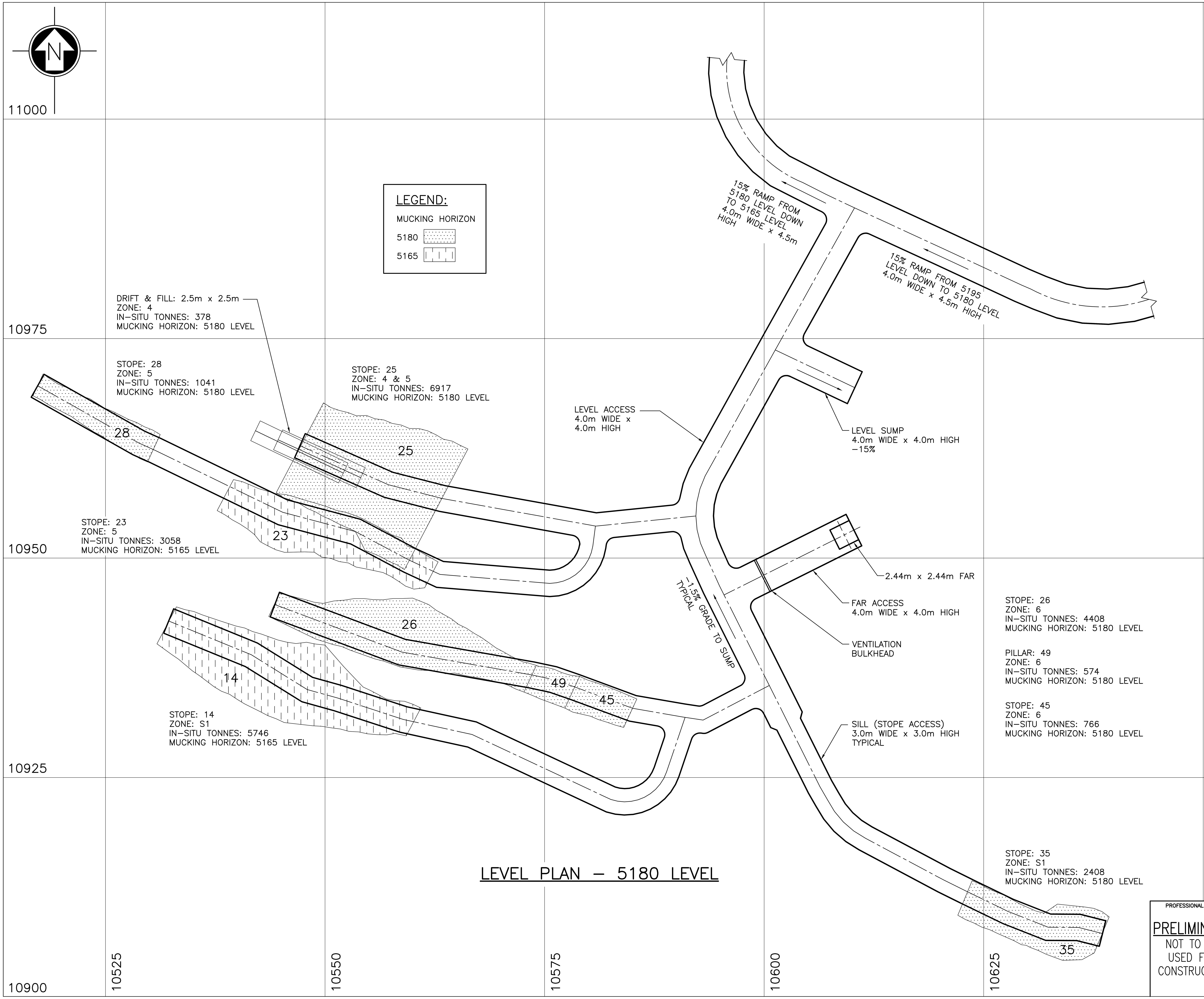

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No	REVISION DESCRIPTION	DATE (Y/M/D)	*	**
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BPB	ISSUED FOR CLIENT REVIEW	16/12/14		
BPA	ISSUED FOR COMMENTS	16/11/15		

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SCALE 1:200	

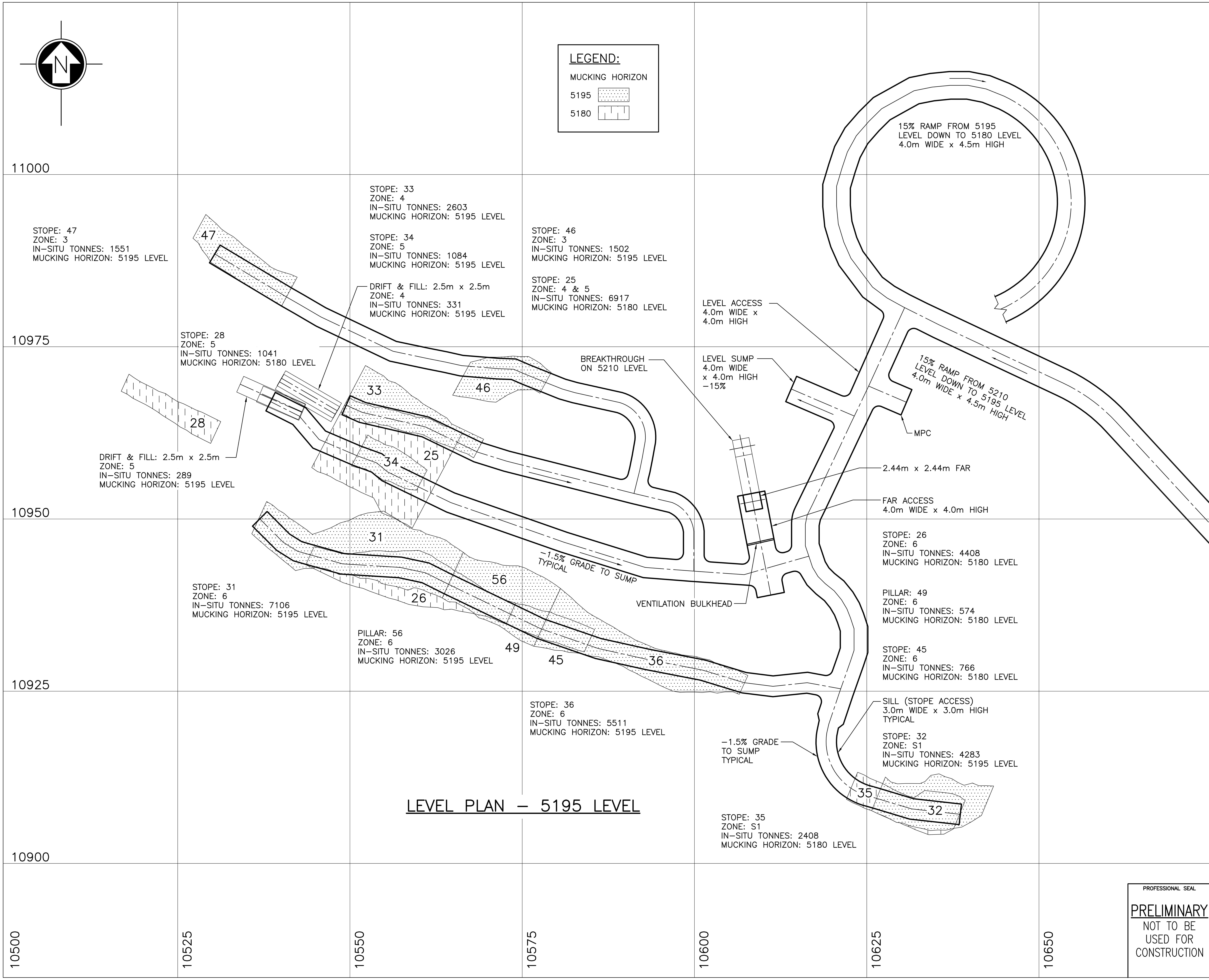
CLIENT
WALLBRIDGE MINING COMPANY LTD

PROJECT
FENELON GOLD MINE

TITLE
UNDERGROUND MINE DESIGN
TYPICAL LAYOUT
5180 LEVEL
PLAN

PROJECT No	SUBDIVISION	SUBJECT	SERIAL	REV.
640914	0000	4M DD	0006	B00

LAST SAVE: 2016/12/13 – 5:37pm
PATH: P:\300-RN\640914\0000\0-Docs Emis\2017-01-12 – dessins pour Simon\Level Plans – REV B – ISSUED FOR CLIENT REVIEW\640914-0000-4MDD-0006_B00.dwg



LEVEL PLAN – 5195 LEVEL

WALLBRIDGE MINING COMPANY LIMITED

ISSUE REGISTER

ISSUE No	REV.	DATE (Y/M/D)	PURPOSE OF ISSUE	TRANSMISSION LETTER No

REVISION REGISTER

No	REVISION DESCRIPTION	DATE (Y/M/D)	*	**
B00	ISSUED FOR PRE-FEASIBILITY	17/01/12		
BPB	ISSUED FOR CLIENT REVIEW	16/12/14		
BPA	ISSUED FOR COMMENTS	16/11/15		

INITIALS: * DESIGNED ** APPROVED

No	REFERENCE DRAWINGS NUMBER

SNC·LAVALIN
SNC–LAVALIN INC.
40 Larch Street, Suite 300
Sudbury, Ontario, Canada
P3E 5M7 | 705–222–0164

PREPARATION	APPROVAL
DESIGNED RFL	PROJECT DISCIPLINE ENGINEER
DRAWN RFL	PROJECT ENGINEERING MANAGER
CHECKED GB	
DATE 16/11/07	CLIENT
SCALE 1:250	

CLIENT
WALLBRIDGE MINING COMPANY LTD

PROFESSIONAL SEAL
PRELIMINARY
NOT TO BE USED FOR CONSTRUCTION

PROJECT
FENELON GOLD MINE

TITLE
UNDERGROUND MINE DESIGN
LEVEL LAYOUT
5195 LEVEL
PLAN

PROJECT No	SUBDIVISION	SUBJECT	SERIAL	REV.
640914	0000	4M DD	0007	B00

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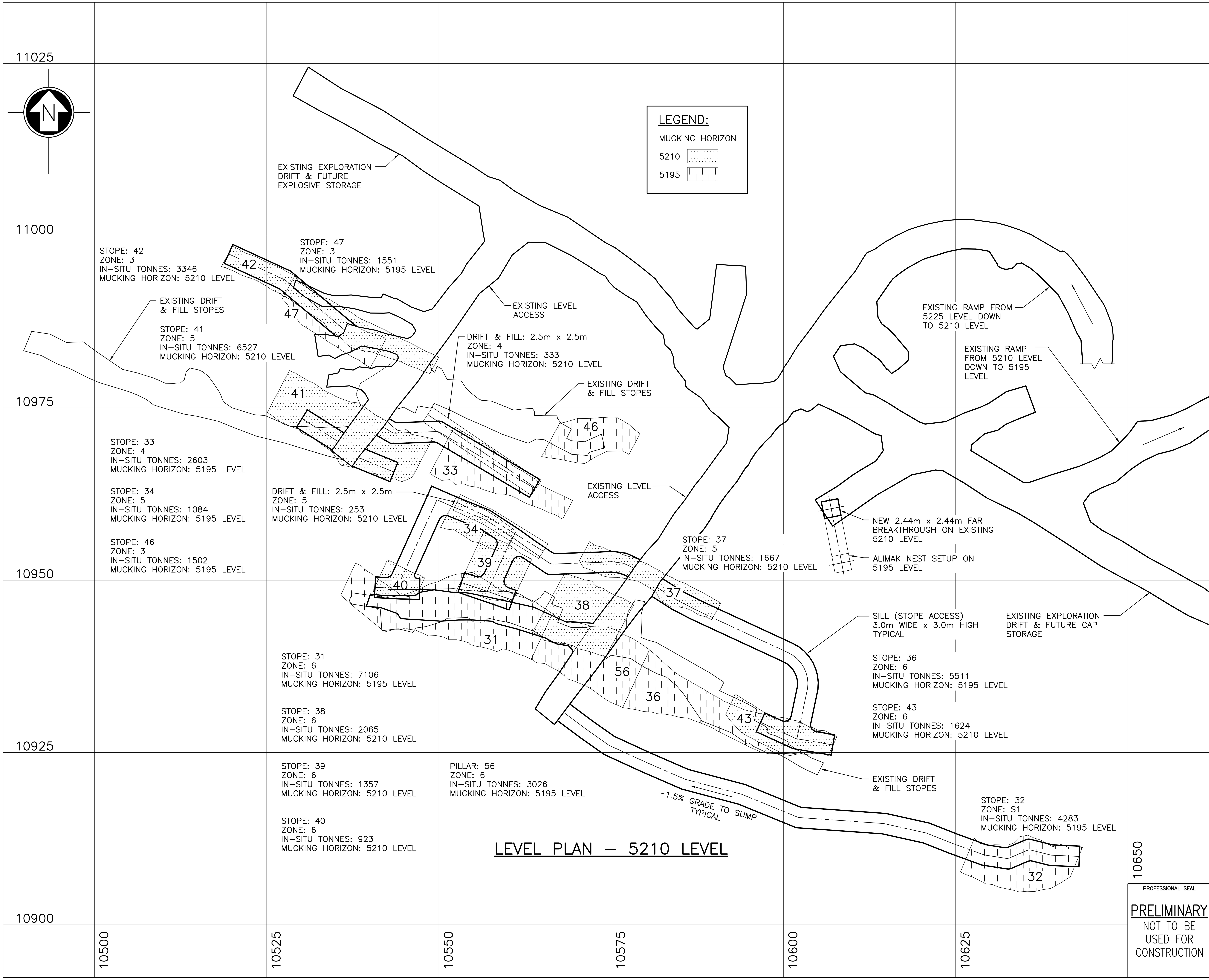

LAST SAVE: 2016/12/13 – 5:39pm
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WALLBRIDGE MINING COMPANY LIMITED

ISSUE REGISTER

ISSUE No	REV.	DATE (Y/M/D)	PURPOSE OF ISSUE	TRANSMISSION LETTER No

No	REVISION DESCRIPTION	DATE (Y/M/D)	*	**
B00	ISSUED FOR PRE-FEASIBILITY	17/01/12		
BPB	ISSUED FOR CLIENT REVIEW	16/12/14		
BPA	ISSUED FOR COMMENTS	16/11/15		

INITIALS: * DESIGNED ** APPROVED

REVISION REGISTER

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SNC·LAVALIN
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PREPARATION	APPROVAL
DESIGNED RFL	PROJECT DISCIPLINE ENGINEER
DRAWN RFL	PROJECT ENGINEERING MANAGER
CHECKED GB	
DATE 16/11/07	CLIENT
SCALE 1:250	

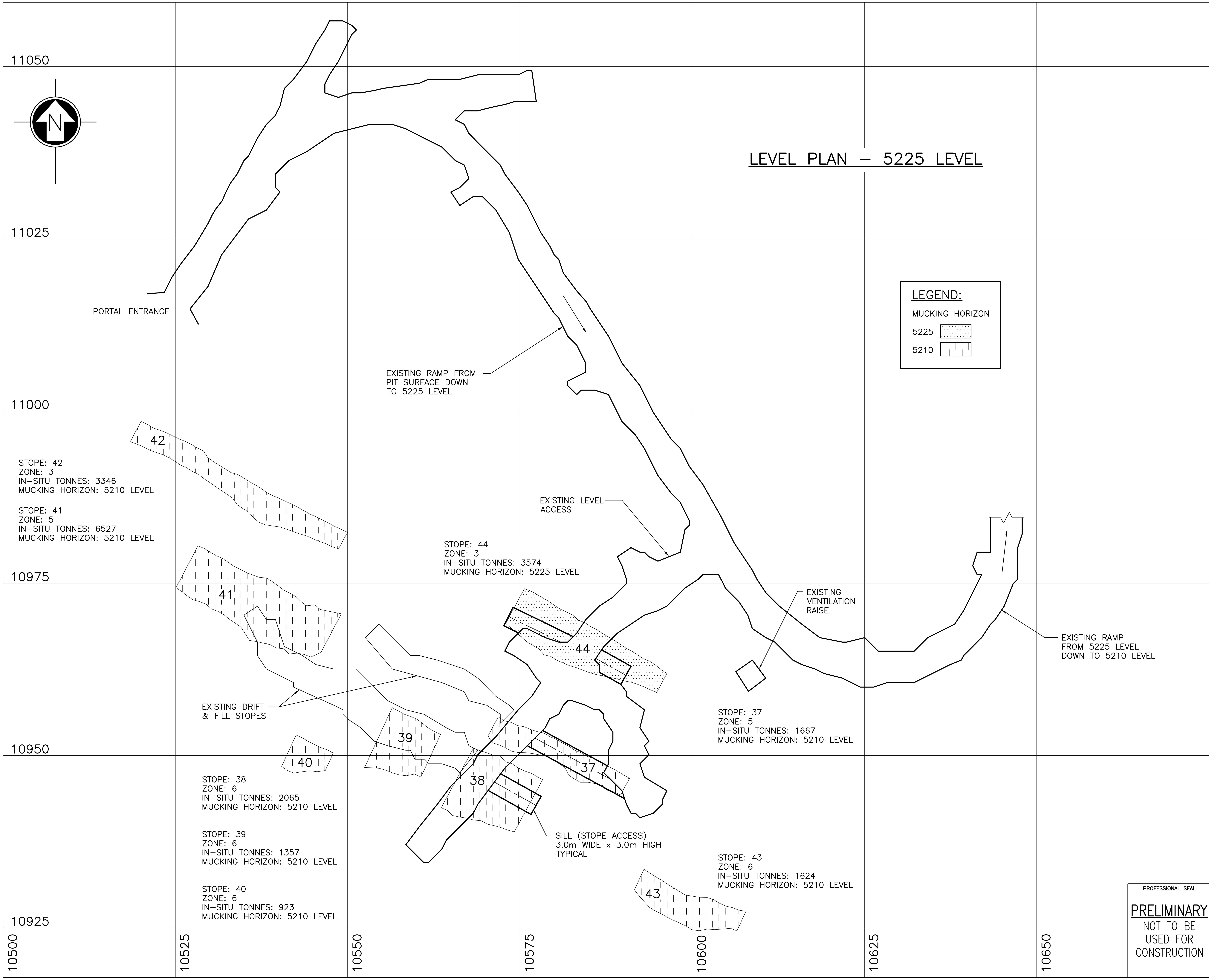
CLIENT
WALLBRIDGE MINING COMPANY LTD

PROFESSIONAL SEAL
PRELIMINARY
NOT TO BE USED FOR CONSTRUCTION

PROJECT
FENELON GOLD MINE

TITLE
UNDERGROUND MINE DESIGN
LEVEL LAYOUT
5210 LEVEL
PLAN

PROJECT No	SUBDIVISION	SUBJECT	SERIAL	REV.
640914	0000	4M DD	0008	B00



WALLBRIDGE
MINING COMPANY LIMITED

ISSUE REGISTER

ISSUE No	REV.	DATE (Y/M/D)	PURPOSE OF ISSUE	TRANSMISSION LETTER No

No	REVISION DESCRIPTION	DATE (Y/M/D)	*	**
B00	ISSUED FOR PRE-FEASIBILITY	17/01/12		
BPB	ISSUED FOR CLIENT REVIEW	16/12/14		
BPA	ISSUED FOR COMMENTS	16/11/15		

INITIALS: * DESIGNED ** APPROVED

REVISION REGISTER

No	REFERENCE DRAWINGS NUMBER

SNC·LAVALIN
SNC–LAVALIN INC.
40 Larch Street, Suite 300
Sudbury, Ontario, Canada
P3E 5M7 | 705–222–0164

PREPARATION	APPROVAL
DESIGNED RFL	PROJECT DISCIPLINE ENGINEER
DRAWN RFL	PROJECT ENGINEERING MANAGER
CHECKED GB	
DATE 16/11/14	CLIENT
SCALE 1:250	

PROFESSIONAL SEAL

PRELIMINARY
NOT TO BE USED FOR CONSTRUCTION

CLIENT
WALLBRIDGE MINING COMPANY LTD

PROJECT
FENELON GOLD MINE

TITLE
UNDERGROUND MINE DESIGN
LEVEL LAYOUT
5225 LEVEL
PLAN

PROJECT No	SUBDIVISION	SUBJECT	SERIAL	REV.
640914	0000	4M DD	0009	B00

LAST SAVE: 2016/12/13 – 5:41pm
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APPENDIX VI – COST ESTIMATE

WALLBRIDGE MINING COMPANY LIMITED

Cost Estimate Detail
Date: 17-01-2017
Revision: B00

SNC·LAVALIN

Financial Type Code	Financial Type Description	Main Area Code	Main Area Description	Wallbridge Cashflow	Wallbridge Cashflow Description	Wallbridge Cashflow Code and Description	Sub Area Code	SubArea Description	Order	System Code	System Description	Estimate Description	UoM	Qty	Engineering Growth	Final Qty	Man hours	PF	Total Hours	Labour Rate	Total Labour	Material Unit Rate	Total Material	Equipment Unit Rate	Total Equipment	Subtrade Unit Rate	Total Subtrade	Total	Unit Rate
1	Direct	1	Surface	1	Preproduction	1 : Preproduction	1	Surface	3	70	Salbaie to mine access road repairs	Salbaie to mine access road repairs	km	0	1	0		1.3	0	90	0		0		0	15000	0	0	0
1	Direct	9	Indirect	1	Preproduction	1 : Preproduction	9	Project Indire	4	21	Engineering	Execution Engineering	ls	1	1	1			0		0		0		0	400000	400000	400000	400000
1	Preproductior	9	Indirect	1	Preproduction	1 : Preproduction	9	Opex	5	65	Closure	Closure	ls	1	1	1			0		0		0		0	896217	896217	896217	896217
2	Indirect	9	Indirect	1	Preproduction	1 : Preproduction	9	Project Indire	2	04	Permits and Approvals	Permits and Approval	ls	1	1	1			0		0		0		0	200000	200000	200000	200000
1	Direct	1	Surface	2	Capital	2 : Capital	1	Surface	10	01	Mine Access Road Upgrade	Upgrade mine access road	km	5	1	5			0		0		0		0	50000	250000	250000	50000
1	Direct	1	Surface	2	Capital	2 : Capital	1	Surface	9	02	Polishing Pond	Excavation for Settling Pond	m3	2352	1	2352	0.014	1.3	42.8064	90	3852.576		0		0	20	47040	50892.576	21.638
1	Direct	1	Surface	2	Capital	2 : Capital	1	Surface	9	02	Polishing Pond	Construction of clay berms	m3	2160	1	2160	0.03	1.3	84.24	90	7581.6	40	86400		0		0	93981.6	43.51
1	Direct	1	Surface	2	Capital	2 : Capital	1	Surface	9	02	Polishing Pond	Supply and place GSC membrane	m2	170	1	170	0.03	1.3	6.63	90	596.7	8	1360		0		0	1956.7	11.51
1	Direct	1	Surface	2	Capital	2 : Capital	1	Surface	9	02	Polishing Pond	Supply and place Geotextile filter cloth	m2	170	1	170	0.03	1.3	6.63	90	596.7	2	340		0		0	936.7	5.51
1	Direct	1	Surface	2	Capital	2 : Capital	1	Surface	9	02	Polishing Pond	Rip Rap for berm erosion control	m3	588	1	588	0.5	1.3	382.2	90	34398	309	181692		0		0	216090	367.5
3	Opex	2	Undergrounc	3	Operating	3 : Operating	2	5225 Level	13	14	Underground Development Sills	Sill Development 3.0m x 3.0m	m	19.8	1	20			0		0		0		0	1924	38095.2	38095.2	1924
1	Direct	2	Undergrounc	2	Capital	2 : Capital	2	5225 Level		14	Underground Development Rehabilitate Ex	Rehabilitation of existing development	ls	1	1	1			0		0		0		0	377100	377100	377100	377100
1	Direct	2	Undergrounc	2	Capital	2 : Capital	3	5210 Level	12	14	Underground Development Level Access	Lateral Development 4.0m x 4.0m	m	33.4	1	33			0		0		0		0	2467	82397.8	82397.8	2467
3	Opex	2	Undergrounc	3	Operating	3 : Operating	3	5210 Level	13	14	Underground Development Sills	Sill Development 3.0m x 3.0m	m	180.4	1	180			0		0		0		0	1924	347089.6	347089.6	1924
1	Direct	2	Undergrounc	2	Capital	2 : Capital	4	5195 Level	11	14	Underground Development Ramp	Ramp Development 4.5m x 4.0 m	m	100.8	1	101			0		0		0		0	3176	320140.8	320140.8	3176
1	Direct	2	Undergrounc	2	Capital	2 : Capital	4	5195 Level	12	14	Underground Development Level Access	Lateral Development 4.0m x 4.0m	m	122.5	1	123			0		0		0		0	2467	302207.5	302207.5	2467
3	Opex	2	Undergrounc	3	Operating	3 : Operating	4	5195 Level	13	14	Underground Development Sills	Sill Development 3.0m x 3.0m	m	200.1	1	200			0		0		0		0	1924	384992.4	384992.4	1924
1	Direct	2	Undergrounc	2	Capital	2 : Capital	4	5195 Level		15	Underground Development Raises	Raise Development 2.4m x 2.4m	vm	17	1	17			0		0		0		0	2661	45237	45237	2661
1	Direct	2	Undergrounc	2	Capital	2 : Capital	4	5195 Level		16	Underground Development Raises	Equip Raise with Manway	vm	14.4	1	14			0		0		0		0	2593	37339.2	37339.2	2593
1	Direct	2	Undergrounc	2	Capital	2 : Capital	5	5180 Level	11	14	Underground Development Ramp	Ramp Development 4.5m x 4.0 m	m	130.9	1	131			0		0		0		0	2908	380657.2	380657.2	2908
1	Direct	2	Undergrounc	2	Capital	2 : Capital	5	5180 Level	12	14	Underground Development Level Access	Lateral Development 4.0m x 4.0m	m	109.4	1	109			0		0		0		0	2467	269889.8	269889.8	2467
3	Opex	2	Undergrounc	3	Operating	3 : Operating	5	5180 Level	13	14	Underground Development Sills	Sill Development 3.0m x 3.0m	m	187.9	1	188			0		0		0		0	1924	361519.6	361519.6	1924
1	Direct	2	Undergrounc	2	Capital	2 : Capital	5	5180 Level		15	Underground Development Raises	Raise Development 2.4m x 2.4m	vm	11	1	11			0		0		0		0	2661	29271	29271	2661
1	Direct	2	Undergrounc	2	Capital	2 : Capital	5	5180 Level		16	Underground Development Raises	Equip Raise with Manway	vm	14.4	1	14			0		0		0		0	2593	37339.2	37339.2	2593
1	Direct	2	Undergrounc	2	Capital	2 : Capital	6	5165 Level	11	14	Underground Development Ramp	Ramp Development 4.5m x 4.0 m	m	117.9	1	118			0		0		0		0	2908	342853.2	342853.2	2908
1	Direct	2	Undergrounc	2	Capital	2 : Capital	6	5165 Level	12	14	Underground Development Level Access	Lateral Development 4.0m x 4.0m	m	103.7	1	104			0		0		0		0	2467	255827.9	255827.9	2467
3	Opex	2	Undergrounc	3	Operating	3 : Operating	6	5165 Level	13	14	Underground Development Sills	Sill Development 3.0m x 3.0m	m	97.3	1	97			0		0		0		0	1924	187205.2	187205.2	1924
1	Direct	2	Undergrounc	2	Capital	2 : Capital	6	5165 Level		15	Underground Development Raises	Raise Development 2.4m x 2.4m	vm	11	1	11			0		0		0		0	2661	29271	29271	2661
1	Direct	2	Undergrounc	2	Capital	2 : Capital	6	5165 Level		16	Underground Development Raises	Equip Raise with Manway	vm	11.1	1	11			0		0		0		0	2593	28782.3	28782.3	2593
1	Direct	2	Undergrounc	2	Capital	2 : Capital	7	5150 Level	11	14	Underground Development Ramp	Ramp Development 4.5m x 4.0 m	m	141.6	1	142			0		0		0		0	2908	411772.8	411772.8	2908
1	Direct	2	Undergrounc	2	Capital	2 : Capital	7	5150 Level	12	14	Underground Development Level Access	Lateral Development 4.0m x 4.0m	m	114.5	1	115			0		0		0		0	2467	282471.5	282471.5	2467
3	Opex	2	Undergrounc	3	Operating	3 : Operating	7	5150 Level	13	14	Underground Development Sills	Sill Development 3.0m x 3.0m	m	38.7	1	39			0		0		0		0	1924	74458.8	74458.8	1924
1	Direct	2	Undergrounc	2	Capital	2 : Capital	7	5150 Level		15	Underground Development Raises	Raise Development 2.4m x 2.4m	vm	11	1	11			0		0		0		0	2661	29271	29271	2661
1	Direct	2	Undergrounc	2	Capital	2 : Capital	7	5150 Level		16	Underground Development Raises	Equip Raise with Manway	vm	11.1	1	11			0		0		0		0	2593	28782.3	28782.3	2593
1	Direct	2	Undergrounc	3	Operating	3 : Operating	0	General	16	50	Delineation Drilling	Delineation Drilling	m	3000	1	3000			0		0		0		0	109	327000	327000	109
1	Direct	1	Surface	2	Capital	2 : Capital	1	Surface		02	Dewatering	Dewatering of Open Pit of 225000 m3 of water	ls	1	1	1			0	90	0		0		0	182500	182500	182500	182500
3	Opex	1	Surface	2	Capital	2 : Capital	64	Construction Indirect		14	Mining Contractor Mobilization	Mobilization of Mining Contractor	ls	1	1	1			0		0		0		0	411340	411340	411340	411340
3	Opex	1	Surface	3	Operating	3 : Operating	10	Opex		05	Crushing	Mobilization of Crushing Contractor	ls	1	1	1			0		0		0		0	26000	26000	26000	26000
3	Opex	1	Surface	2	Capital	2 : Capital	10	Opex		03	Site Setup	Surface Setup : Contractor Temporary buildings	ls	1	1	1			0		0		0		0	622400	622400	622400	622400
1	Direct	1	Surface	3	Operating	3 : Operating	1	Surface	15	12	Ventilation	Surface Setup : Fans and heater: 125,000 CFM a	ls	1	1	1			0		0		0	71200	71200		0	71200	71200
2	Indirect	1	Surface	3	Operating	3 : Operating	10	Opex		64	Contractor Indirect	Contractor Indirect Labour -Setup/Mob/TearDo	days	81	1	81			0		0		0		0	9080	735480	735480	9080
2	Indirect	1	Surface	3	Operating	3 : Operating	10	Opex		64	Contractor Indirect	Equipment Operating -Setup/Mob/TearDown: I	days	81	1	81			0		0		0		0	4820	390420	390420	4820
1	Direct	2	Undergrounc	3	Operating	3 : Operating	2	5225 Level		12	Ventilation	Supply and install 36" 50 hp fan	ls	1	1	1			0		0		0	24328	24328		0	24328	24328
1	Direct	2	Undergrounc	3	Operating	3 : Operating	3	5210 Level		12	Ventilation	Supply and install Regulator with 36" 50 hp fan	ls	1	1	1			0		0		0	42155	42155		0	42155	42155
1	Direct	2	Undergrounc	3	Operating	3 : Operating	3	5210 Level		04	Dewatering	Supply and construct level sump with 50 Hp Pur	ls	1	1	1			0		0		0	32576	32576		0	32576	32576
1	Direct	2	Undergrounc	2	Capital	2 : Capital	3	5210 Level		04	Site Setup	Supply and construct latrine	ls	1	1	1			0		0		0	10838	10838		0	10838	10838
1	Direct	2	Undergrounc	2	Capital	2 : Capital	3	5210 Level		04	Site Setup	Supply and construct refuge station	ls	1	1	1			0		0		0	93176	93176		0	93176	93176
1	Direct	2	Undergrounc	2	Capital	2 : Capital	3	5210 Level		04	Site Setup	Supply and construct cap storage	ls	1	1	1			0		0		0	5688	5688		0	5688	5688
1	Direct	2	Undergrounc	2	Capital	2 : Capital	3	5210 Level		04	Site Setup	Supply and construct explosive storage	ls	1	1	1			0		0		0	8026	8026		0	8026	8026
1	Direct	2	Undergrounc	3	Operating	3 : Operating	4	5195 Level		12	Ventilation	Supply and install Regulator with 36" 50 hp fan	ls	1	1	1			0		0		0	42155	42155		0	42155	42155
1	Direct	2	Undergrounc	3	Operating	3 : Operating	4	5195 Level		04	Dewatering	Supply and construct level sump with 20 Hp Pur	ls	1	1	1			0		0		0	19998	19998		0	19998	19998
1	Direct	2	Undergrounc	3	Operating	3 : Operating	5	5180 Level		12	Ventilation	Supply and install Regulator with 36" 50 hp fan	ls	1	1	1			0		0		0	42155	42155		0	42155	42155
1	Direct	2	Undergrounc	3	Operating	3 : Operating	5	5180 Level		04	Dewatering	Supply and construct level sump with 20 Hp Pur	ls	1	1	1			0		0		0	19998	19998		0	19998	19998
1	Direct	2	Undergrounc	3	Operating	3 : Operating	6	5165 Level		12	Ventilation	Supply and install Regulator with 36" 50 hp fan	ls	1	1	1			0		0		0	42155	42155		0	42155	42155
1	Direct	2	Undergrounc	3	Operating	3 : Operating	6	5165 Level		04	Dewatering	Supply and construct level sump with 20 Hp Pur	ls	1	1	1			0		0		0	19998	19998		0	19998	19998
1	Direct	2	Undergrounc	3	Operating	3 : Operating	7	5150 Level		12	Ventilation	Supply and install Regulator with 36" 50 hp fan	ls	1	1	1			0		0		0	42155	42155		0	42155	42155
1	Direct	2	Undergrounc	3	Operating	3 : Operating	6	5165 Level		04	Dewatering	Supply and construct level sump with 20 Hp Pur	ls	1	1	1			0		0		0	19998	19998		0	19998	19998
2	Indirect	2	Undergrounc	3	Operating	3 : Operating	10	Opex		64	Contractor Indirect	Contractor Indirect Staff - Mine Lateral develop	days	105	1	105			0		0		0		0	12510	1313550	1313550	12510
2	Indirect	2	Undergrounc	3	Operating	3 : Operating	10	Opex		64	Contractor Indirect	Indirect Equipment Operating: Mine Lateral Dev	days	105	1	105			0		0		0	8830	927150		0	927150	8830
3	Opex	2	Undergrounc	3	Operating	3 : Operating	2	5225 Level	13	14	Underground Ore Development	Sill Ore 3.0m x 3.0m	m	13.7	1	14			0		0		0		0	1924	26358.8	26358.8	1924
3	Opex	2	Undergrounc	3	Operating	3 : Operating	2	5225 Level		14	Underground Ore Development	Stope Development (Drill/Blast/Muck)	tonnes	3574.4	1	3574			0		0		0		0	20.6	73632.64	73632.64	20.6
3	Opex	2	Undergrounc	3	Operating	3 : Operating	2	5225 Level		14	Underground Ore Development	Slot Raise - Drill and Blast	vm	30	1	30			0		0		0		0	525.1	15753	15753	525.1
3	Opex	1	Surface	3	Operating	3 : Operating	2	5225 Level		67	Crushing	Crush ROM ore to 18" minus	tonnes	3905	1	3905			0		0		0		0	4.6	17963	17963	4.6
3	Opex	1	Surface	3	Operating	3 : Operating	2	5225 Level		68	Transport to Mill	Transport to Camflo Mill	tonnes	3905	1	3905			0		0		0		0	34.13	133277.65	133277.65	34.13
3	Opex	1	Surface	3	Operating	3 : Operating	2	5225 Level		69	Milling	Milling of ore at Camflo Mill	tonnes	3905	1	3905			0		0		0		0	37	144485	144485	37
3	Opex	2	Undergrounc	3	Operating	3 : Operating	3	5210 Level	13	14	Underground Ore Development	Sill Ore 3.0m x 3.0m	m	62.9	1	63			0		0		0		0	1924	121019.6	121019.6	1924
3	Opex	2	Undergrounc	3	Operating	3 : Operating	3	5210 Level		14	Underground Ore Development	Stope Development	tonnes	17933	1	17933			0		0		0		0	20.6	369419.8	369419.8	20.6
3	Opex	2	Undergrounc	3	Operating	3 : Operating	3	5210 Level		14	Underground Ore Development	Slot Raise - Drill and Blast	vm	105	1	105			0		0		0		0	525.1	55135.5	55135.5	525.1
3	Opex	2	Undergrounc	3	Operating	3 : Operating	3	5210 Level		14	Backfill Stope	Backfill stope - Rockfill only	tonnes	10160	1	10160			0		0		0		0	7.04	71526.4	71526.4	7.04
3	Opex	1	Surface	3	Operating	3 : Operating	3	5210 Level		67	Crushing	Crush ROM ore to 18" minus	tonnes	23055	1	23055			0		0		0		0	4.6	106053	106053	4.6
3	Opex	1	Surface	3	Operating	3 : Operating	3	5210 Level		68	Transport to Mill	Transport to Camflo Mill	tonnes	23055	1	23055			0		0		0		0	34.13	786867.15	786867.15	34.13
3	Opex	1	Surface	3	Operating	3 : Operating	3	5210 Level		69	Milling	Milling of ore at Camflo Mill	tonnes	23055	1	23055			0		0		0		0	37	853035	853035	37
3	Opex	2	Undergrounc	3	Operating	3 : Operating	4	5195 Level	13	14	Underground Ore Development	Sill Ore 3.0m x 3.0m	m	133.8	1	134			0		0		0		0	1924	257431.2	257431.2	1924
3	Opex	2	Undergrounc	5	Operating	3 : Operating	4	5195 Level		14	Underground Ore Development	Stope Development	tonnes	25087	1	25087			0		0		0		0	20.6	516792.2	516792.2	20.6
3	Opex	2	Undergrounc	3	Operating	3 : Operating	4	5195 Level		14	Underground Ore Development	Slot Raise - Drill and Blast	vm	104	1	104			0		0		0		0	525.1	54610.4	54610.4	525.1
3	Opex	2	Undergrounc	3	Operating	3 : Operating	4	5195 Level		14	Backfill Stope	Backfill stope - Rockfill only	tonnes	11012	1	11012			0		0		0		0	7.04	77524.48	77524.48	7.04
3	Opex	2	Undergrounc	3	Operating	3 : Operating	4	5195 Level		14	Backfill Stope	Backfill stope - Cemented rockfill	tonnes	3306	1	3306			0		0		0		0	100	330600	330600	100
3	Opex	1	Surface	5	Operating	3 : Operating	4	5195 Level		67	Crushing	Crush ROM ore to 18" minus	tonnes	28932	1	28932			0		0		0		0	4.6	133087.2	133087.2	4.6
3	Opex	1	Surface	5	Operating	3 : Operating	4	5195 Level		68	Transport to Mill	Transport to Camflo Mill	tonnes	28932	1	28932			0		0		0		0	34.13	987449.16	987449.16	34.13
3	Opex	1	Surface		Operating	3 : Operating	4	5195 Level		69	Milling	Milling of ore at Camflo Mill	tonnes	28932	1	28932			0		0		0		0	37	1070484	1070484	37
3	Opex	2	Undergrounc	3	Operating	3 : Operating	5	5180 Level	13	14	Underground Ore Development	Sill Ore 3.0m x 3.0m	m	98.5	1	99			0		0		0		0	1924	189514	189514	1924
3	Opex	2	Undergrounc	3	Operating	3 : Operating	5	5180 Level		14	Underground Ore Development	Stope Development	tonnes	15710	1	15710			0		0		0		0	20.6	323626	323626	20.6
3	Opex	2	Undergrounc	3	Operating	3 : Operating	5	5180 Level		14	Underground Ore Development	Slot Raise - Drill and Blast	vm	54	1	54			0		0		0		0	525.1	28355.4	28355.4	525.1
3	Opex	2	Undergrounc	3	Operating	3 : Operating	5	5180 Level		14	Backfill Stope	Backfill stope - Rockfill only	tonnes	16868	1	16868			0		0		0		0	7.04	118750.72	118750.72	7.04
3	Opex	2	Undergrounc	3	Operating	3 : Operating	5	5180 Level		14	Backfill Stope	Backfill stope - Cemented rockfill	tonnes	641	1	641			0		0		0		0	100	64100	64100	100
3	Opex	1	Surface	3	Operating	3 : Operating	5	5180 Level		67	Crushing	Crush ROM ore to 18" minus	tonnes	18466	1	18466			0		0		0		0	4.6	84943.6	84943.6	4.6
3	Opex	1	Surface	3	Operating	3 : Operating	5	5180 Level		68	Transport to Mill	Transport to Camflo Mill	tonnes	18466	1	18466			0		0		0		0	34.13	630244.58	630244.58	34.13
3	Opex	1	Surface	3	Operating	3 : Operating	5	5180 Level		69	Milling	Milling of ore at Camflo Mill	tonnes	18466	1	18466			0		0		0		0	37	683242	683242	37
3	Opex	2	Undergrounc	3	Operating	3 : Operating	6	5165 Level	13	14	Underground Ore Development	Sill Ore 3.0m x 3.0m	m	66.6	1	67			0		0		0		0	1924	128138.4	128138.4	1924
3	Opex	2	Undergrounc	3	Operating	3 : Operating	6	5165 Level		14	Underground Ore Development	Stope Development	tonnes	8930	1	8930			0		0		0		0	20.6	183958	183958	20.6
3	Opex	2	Undergrounc	3	Operating	3 : Operating	6	5165 Level		14	Underground Ore Development	Slot Raise - Drill and Blast	vm	30	1	30			0		0		0		0	525.1	15753	15753	525.1
3	Opex	2	Undergrounc	3	Operating	3 : Operating	6	5165 Level		14	Backfill Stope	Backfill stope - Rockfill only	tonnes	10257	1	10257			0		0		0		0	7.04	72209.28	72209.28	7.04
3	Opex	1	Surface	3	Operating	3 : Operating	6	5165 Level		67	Crushing	Crush ROM ore to 18" minus	tonnes	10782	1	10782			0		0		0		0	4.6	49597.2	49597.2	4.6
3	Opex	1	Surface	3	Operating	3 : Operating	6	5165 Level		68	Transport to Mill	Transport to Camflo Mill	tonnes	10782	1	10782			0		0		0		0	34.13	367989.66	367989.66	34.13
3	Opex	1	Surface	3	Operating	3 : Operating	6	5165 Level		69	Milling	Milling of ore at Camflo Mill	tonnes	10782	1	10782			0		0		0		0	37	398934	398934	37
3	Opex	2	Undergrounc	3	Operating	3 : Operating	7	5150 Level	13	14	Underground Ore Development	Sill Ore 3.0m x 3.0m	m	52.5	1	53			0		0		0		0	1924	101010	101010	1924
3	Opex	2	Undergrounc	3	Operating	3 : Operating	7	5150 Level		14	Underground Ore Development	Stope Development	tonnes	7436	1	7436			0		0		0		0	20.6	153181.6	153181.6	20.6
3	Opex	2	Undergrounc	3	Operating	3 : Operating	7	5150 Level		14	Underground Ore Development	Slot Raise - Drill and Blast	vm	36	1	36			0		0		0		0	525.1	18903.6	18903.6	525.1
3	Opex	2	Undergrounc	3	Operating	3 : Operating	7	5150 Level		14	Backfill Stope	Backfill stope - Rockfill only	tonnes	7785	1	7785			0		0		0		0	7.04	54806.4	54806.4	7.04
3	Opex	2	Undergrounc	3	Operating	3 : Operating	7	5150 Level		14	Backfill Stope	Backfill stope - Cemented rockfill	tonnes	1317	1	1317			0		0		0		0	100	131700	131700	100
3	Opex	1	Surface	3	Operating	3 : Operating	7	5150 Level		67	Crushing	Crush ROM ore to 18" minus	days	9102	1	9102			0		0		0		0	4.6	41869.2	41869.2	4.6
3	Opex	1	Surface	3	Operating	3 : Operating	7	5150 Level		68	Transport to Mill	Transport to Camflo Mill	tonnes	9102	1	9102			0		0		0		0	34.13	310651.26	310651.26	34.13
3	Opex	1	Surface	3	Operating	3 : Operating	7	5150 Level		69	Milling	Milling of ore at Camflo Mill	tonnes	9102	1	9102			0		0		0		0	37	336774	336774	37
3	Opex	2	Undergrounc	3	Operating	3 : Operating	11	Pit		14	Underground Ore Development	Stope Development	tonnes	2478	1	2478			0		0		0		0	20.6	51046.8	51046.8	20.6
3	Opex	2	Undergrounc	3	Operating	3 : Operating	6	Pit		67	Crushing	Crush ROM ore to 18" minus	tonnes	2478	1	2478			0		0		0		0	4.6	11398.8	11398.8	4.6
3	Opex	2	Undergrounc	3	Operating	3 : Operating	7	Pit		68	Transport to Mill	Transport to Camflo Mill	tonnes	2478	1	2478			0		0		0		0	34.13	84574.14	84574.14	34.13
3	Opex	2	Undergrounc	3	Operating	3 : Operating	7	Pit		69	Milling	Milling of ore at Camflo Mill	tonnes	2478	1	2478			0		0		0		0	37	91686	91686	37
2	Indirect	2	Undergrounc	3	Operating	3 : Operating	10	Opex		64	Contractor Indirect	Contractor Indirect Staff - Mine Lateral develop	days	46	1	46			0		0		0		0	12510	575460	575460	12510
2	Indirect	2	Undergrounc	3	Operating	3 : Operating	10	Opex		64	Contractor Indirect	Indirect Equipment Operating: Mine Lateral Dev	days	46	1	46			0		0		0	9110	419060		0	419060	9110
2	Indirect	2	Undergrounc	3	Operating	3 : Operating	10	Opex		64	Contractor Indirect	Contractor Indirect Staff - Stoping - 15 man per	days	248	1	248			0		0		0		0	13070	3241360	3241360	13070
2	Indirect	2	Undergrounc	3	Operating	3 : Operating	10	Opex		64	Contractor Indirect	Indirect Equipment Operating Stoping (phase 4)	days	248	1	248			0		0		0	11940	2961120		0	2961120	11940
2	Indirect	1	Surface	1	Preproduction	3 : Operating	4	Opex		65	Site Demobilization	Demobilization of Dorms	ls	1	1	1			0		0		0		0	0	0	0	0
2	Indirect	1	Surface	1	Preproduction	3 : Operating	4	Opex		65	Site Demobilization	Demobilization of Kitchen Units	ea	1	1	1			0		0		0		0	0	0	0	0
2	Indirect	1	Surface	1	Preproduction	3 : Operating	4	Opex		65	Site Demobilization	Demobilization of Seating and Recreation Unit	ea	1	1	1			0		0		0		0	0	0	0	0
2	Indirect	1	Surface	1	Preproduction	3 : Operating	4	Opex		65	Site Demobilization	Demobilization of Administration Building	ls	1	1	1			0		0		0		0	0	0	0	0
2	Indirect	1	Surface	1	Preproduction	3 : Operating	4	Opex		65	Site Demobilization	Demobilization of dry Building	ls	1	1	1			0		0		0		0	0	0	0	0
2	Indirect	1	Surface	1	Preproduction	3 : Operating	4	Opex		65	Site Demobilization	Demobilization of Gate House	ls	1	1	1			0		0		0		0	0	0	0	0
2	Indirect	2	Undergrounc	1	Preproduction	3 : Operating	10	Opex		65	Site Tear Down	Teardown Underground	ls	1	1	1			0		0		0		0	0	0	0	0
3	Opex	1	Surface	1	Preproduction	3 : Operating	10	Opex		65	Site Tear Down	Teardown Surface	ls	1	1	1			0		0		0		0	0	0	0	0
3	Opex	1	Surface	1	Preproduction	3 : Operating	10	Opex		66	Site Demobilization	Site Demobilization	ls	1	1	1			0		0		0		0	0	0	0	0
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		73	Camp Catering and Janitorial	Daily Man Cost - Catering, housekeeping and Lig	man day	1 800	1	1800			0		0	59	106200		0		0	$ 106 200.00	59
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		73	Camp Catering and Janitorial	Daily Man Cost - Catering, housekeeping and Lig	man day	3 150	1	3150			0		0	59	185850		0		0	$ 185 850.00	59
3	Opex	2	Undergrounc	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		73	Camp Catering and Janitorial	Daily Man cost - Catering, housekeeping, and lig	man day	3600	1	3600			0		0	59	212400		0		0	$ 212 400.00	59

WALLBRIDGE MINING COMPANY LIMITED

Cost Estimate Detail
Date: 17-01-2017
Revision: B00

SNC·LAVALIN

Financial Type Code	Financial Type Description	Main Area Code	Main Area Description	Wallbridge Cashflow	Wallbridge Cashflow Description	Wallbridge Cashflow Code and Description	Sub Area Code	SubArea Description	Order	System Code	System Description	Estimate Description	UoM	Qty	Engineering Growth	Final Qty	Man hours	PF	Total Hours	Labour Rate	Total Labour	Material Unit Rate	Total Material	Equipment Unit Rate	Total Equipment	Subtrade Unit Rate	Total Subtrade	Total	Unit Rate
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		73	Camp Catering and Janitorial	Daily Man Cost - Catering, housekeeping and Lig	man day	3 600	1	3600			0		0	59	212400		0		0	$ 212 400.00	59
3	Opex	2	Undergrounc	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		73	Camp Catering and Janitorial	Daily Man cost - Catering, housekeeping, and lig	man day	2880	1	2880			0		0	59	169920		0	0	0	$ 169 920.00	59
3	Opex	2	Undergrounc	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		73	Camp Catering and Janitorial	Daily Man cost - Catering, housekeeping, and lig	man day	900	1	900			0		0	59	53100		0	0	0	$ 53 100.00	59
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		73	Camp Catering and Janitorial	Daily Man cost - Catering, housekeeping, and lig	man day	0.0001	1	0			0		0	59	0.0059		0		0	$ 0.01	59
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		73	Camp Catering and Janitorial	Daily Man cost - Catering, housekeeping, and lig	man day	0.0001	1	0			0		0	59	0.0059		0		0	$ 0.01	59
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Monthly Cost	Rental of 44 person Skidded Dorm	Month	14	1	14			0		0		0	12500	175000		0	175000	12500
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Transport of Dorms from Timmins to site	ea	8	1	8			0		0		0		0	2500	20000	20000	2500
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Transport of Dorms from site to Timmins	ea	8	1	8			0		0		0		0	0	0	0	0
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Installation of Dorms	ea	8	1	8			0		0		0		0	12500	100000	100000	12500
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Provide propane line from bulk tank to dorm	ea	1	1	1			0		0		0		0	3400	3400	3400	3400
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Rough in water and sewer lines for each dorm	ea	1	1	1			0		0		0		0	10200	10200	10200	10200
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Provide buried power distribution to Dorm traile	ea	1	1	1			0		0		0		0	10200	10200	10200	10200
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Monthly Cost	Rental of 2 x 24x60 Kitchen unit	Month	14	1	14			0		0		0	8500	119000		0	119000	8500
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Transport of Kitchen from Montreal to site	ea	2	1	2			0		0		0		0	4200	8400	8400	4200
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Transport of Kitchen from Site to Montreal	ea	2	1	2			0		0		0		0	0	0	0	0
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Installation of 2 x Kitchern Units	ea	1	1	1			0		0		0		0	30000	30000	30000	30000
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Provide propane line from bulk tank to Kitchen	ea	1	1	1			0		0		0		0	3400	3400	3400	3400
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Rough in water and sewer lines for Kitchen	ea	1	1	1			0		0		0		0	10200	10200	10200	10200
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Provide buried power distribution to Kitchen	ea	1	1	1			0		0		0		0	10200	10200	10200	10200
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Monthly Cost	Rental of 1 x 12x60 Seating and Recreation Unit	Month	14	1	14			0		0		0	800	11200		0	11200	800
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Transport of Seating andRecreation Unit to Site	ea	1	1	1			0		0		0		0	2500	2500	2500	2500
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Transport of Seating andRecreation Unit to Tim	ea	1	1	1			0		0		0		0	0	0	0	0
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Installation of Seating and Recreation Unit	ea	1	1	1			0		0		0		0	10000	10000	10000	10000
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Provide propane line from bulk tank to Seating a	ea	1	1	1			0		0		0		0	3400	3400	3400	3400
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Provide buried power distribution to Seating an	ea	1	1	1			0		0		0		0	10200	10200	10200	10200
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Water and Septic ystem	ls	1	1	1			0		0		0	190000	190000		0	190000	190000
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Monthly Cost	Rental of Adminstration Building 12x60 trailer	Month	14	1	14			0		0		0	1900	26600		0	26600	1900
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Transport of Adminstration to site	ls	1	1	1			0		0		0		0	2400	2400	2400	2400
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Transport of Adminstration to Timmins	ls	1	1	1			0		0		0		0	0	0	0	0
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Installation of Adminstration Building	ls	1	1	1			0		0		0		0	10000	10000	10000	10000
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Provide propane line from bulk tank to Adminst	ea	1	1	1			0		0		0		0	3400	3400	3400	3400
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Provide buried power distribution to Adminstra	ea	1	1	1			0		0		0		0	10200	10200	10200	10200
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Transport of dry to site	ls	1	1	1			0		0		0		0	4800	4800	4800	4800
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Transport of dry to home	ls	1	1	1			0		0		0		0	0	0	0	0
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Installation of dry Building	ls	1	1	1			0		0		0		0	30000	30000	30000	30000
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Provide propane line from bulk tank to dry Builc	ea	1	1	1			0		0		0		0	3400	3400	3400	3400
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Provide buried power distribution to dry Buildin	ea	1	1	1			0		0		0		0	10200	10200	10200	10200
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Monthly Cost	Rental of Purchase Gate House Building	Month	14	1	14			0		0		0	750	10500		0	10500	750
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Transportation of Gate House to Site	ls	1	1	1			0		0		0		0	2400	2400	2400	2400
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Transportation of Gate House to Timmins	ls	1	1	1			0		0		0		0	0	0	0	0
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Site up and Demobilization	Installation of Gate House	ls	1	1	1			0		0		0		0	10000	10000	10000	10000
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Monthly Cost	Rental of Generator to power offices and camp	Month	14	1	14			0		0		0	42000	588000		0	588000	42000
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Monthly Cost	Propane for heating of offices and camp	Month	14	1	14			0		0		0	5000	70000		0	70000	5000
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Monthly Cost	Rental of Dry	Month	14	1	14			0		0		0	4900	68600		0	68600	4900
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		72	Camp Monthly Cost	Rental of all inclusive balmoral camp	Month	2	1	2			0		0		0	34860	69720		0	69720	34860
3	Opex	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		73	Camp Catering and Janitorial	Daily Man cost - Catering, housekeeping, and lig	man day	0.0001	1	0			0		0	59	0.0059		0		0	$ 0.01	59
1	Direct	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		01	Road and Site Maintenance	Road and Site Maintenance	Month	18	1	18			0		0		0		0	11614	209052	209052	11614
1	Direct	1	Surface	4	Remote Camp Operation	4 : Remote Camp Operation	4	Remote Camp Operation		01	Salbaie Road Maintenance	Salbaie Road Maintenance	km	90	1	90			0		0		0		0	2717.7	244593	244593	2717.7
3	Opex	1	Surface	5	General and Admission	5 : General and Admission	10	Opex		25	Owner's	Owner's Indirect Staff: Dewatering phase and Si	days	74	1	74			0		0		0		0	6320	467680	467680	6320
3	Opex	2	Undergrounc	5	General and Admission	5 : General and Admission	10	Opex		25	Owner's	Owner's Indirect Staff: Mine Development	days	41	1	41			0		0		0		0	6320	259120	259120	6320
3	Opex	2	Undergrounc	5	General and Admission	5 : General and Admission	10	Opex		25	Owner's	Owner's Indirect Staff: Mine Development and S	days	182	1	182			0		0		0		0	6320	1150240	1150240	6320
3	Opex	2	Undergrounc	5	General and Admission	5 : General and Admission	10	Opex		25	Owner's	Owner's Indirect Staff: Stoping	days	128	1	128			0		0		0		0	6320	808960	808960	6320
3	Opex	9	Indirect	5	General and Admission	5 : General and Admission	10	Opex		26	First Native Recurring	Native First Nation Recurring	Month	18	1	18			0		0		0		0	10000	180000	180000	10000

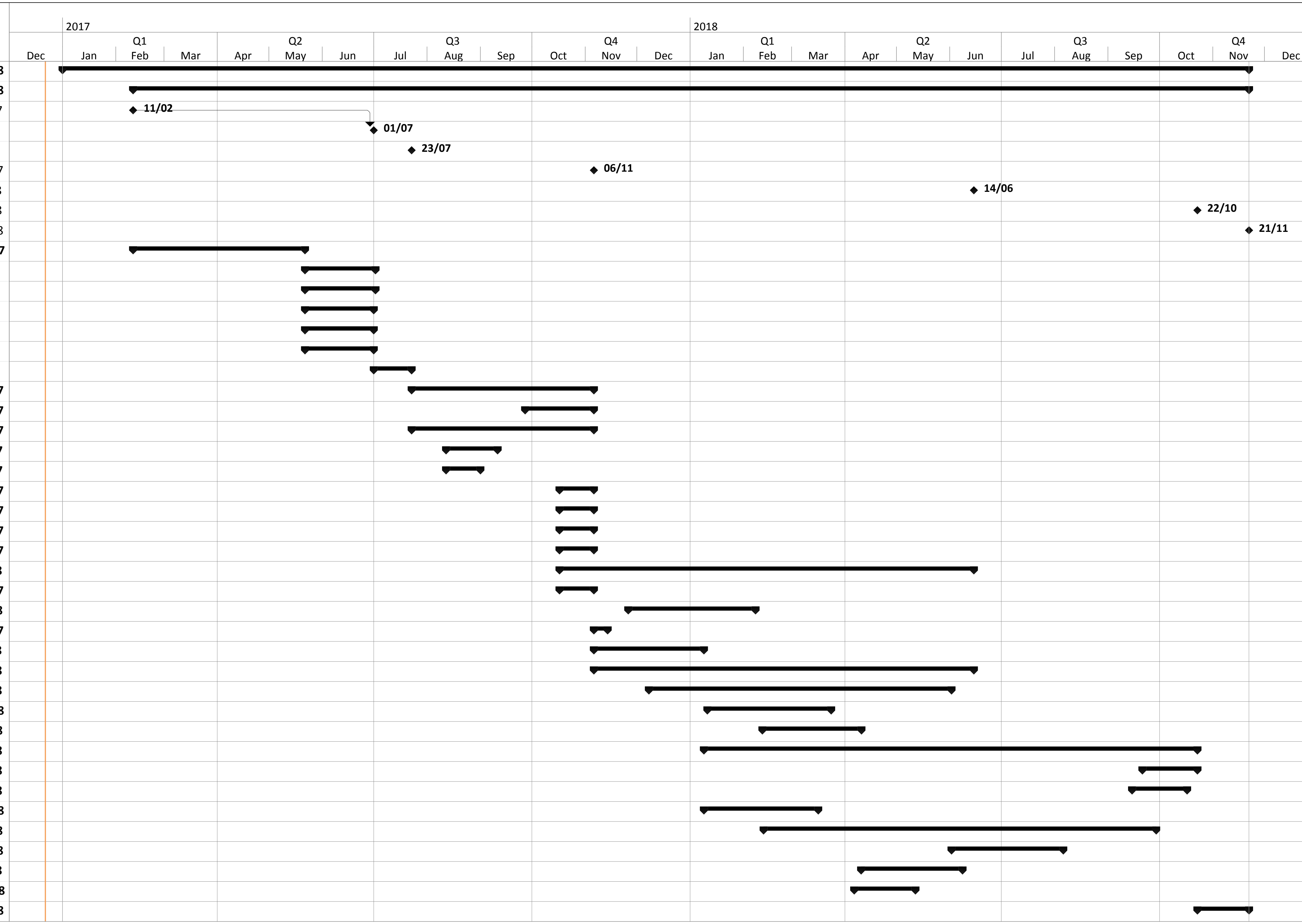
APPENDIX VII – PROJECT AND MINE SCHEDULE

SNC·LAVALIN

FENELON GOLD MINE
PRE-FEASIBILITY STUDY SCHEDULE

WALLBRIDGE
MINING COMPANY LIMITED

ID	Task Name	Duration	Start	Finish
1	**Total Project**	**689.62 days**	**01 Jan '17**	**21 Nov '18**
2	**Milestone**	**648.62 days**	**11 Feb '17**	**21 Nov '18**
3	Engineering Start	0 days	11 Feb '17	11 Feb '17
4	Permits are received	0 days	01 Jul '17	01 Jul '17
5	Mobilization Complete	0 days	23 Jul '17	23 Jul '17
6	Dewatering Complete	0 days	06 Nov '17	06 Nov '17
7	Mine Development Complete	0 days	14 Jun '18	14 Jun '18
8	End of Mining	0 days	22 Oct '18	22 Oct '18
9	Project Finish	0 days	21 Nov '18	21 Nov '18
10	**Engineering**	**100 days**	**11 Feb '17**	**22 May '17**
14	**Procurement**	**41 days**	**22 May '17**	**02 Jul '17**
15	**Proposal Preparation**	**41 days**	**22 May '17**	**02 Jul '17**
18	**Transportation Contract**	**40 days**	**22 May '17**	**01 Jul '17**
21	**Camp contract**	**40 days**	**22 May '17**	**01 Jul '17**
24	**Milling Contract**	**40 days**	**22 May '17**	**01 Jul '17**
27	**Mobilization**	**22 days**	**01 Jul '17**	**23 Jul '17**
31	**Surface**	**106 days**	**23 Jul '17**	**06 Nov '17**
32	**Roads**	**40 days**	**27 Sep '17**	**06 Nov '17**
34	**Basin and Drainage**	**106 days**	**23 Jul '17**	**06 Nov '17**
37	**Camp**	**30 days**	**12 Aug '17**	**11 Sep '17**
40	**Process Water**	**20 days**	**12 Aug '17**	**01 Sep '17**
42	**Ore & Waste Handling - Crusher**	**20 days**	**17 Oct '17**	**06 Nov '17**
44	**Ventilation**	**20 days**	**17 Oct '17**	**06 Nov '17**
46	**Maintenance Shop**	**20 days**	**17 Oct '17**	**06 Nov '17**
48	**Storage & Warehouse**	**20 days**	**17 Oct '17**	**06 Nov '17**
50	**Underground Mine**	**240.52 days**	**17 Oct '17**	**14 Jun '18**
51	**Mine rehabilitation**	**4.67 days**	**17 Oct '17**	**06 Nov '17**
53	**Definition Drilling**	**17.67 days?**	**26 Nov '17**	**07 Feb '18**
55	**5225 Level**	**7.59 days**	**06 Nov '17**	**13 Nov '17**
62	**5210 Level**	**63.46 days**	**06 Nov '17**	**08 Jan '18**
83	**5195 Level**	**220.52 days**	**06 Nov '17**	**14 Jun '18**
114	**5180 Level**	**176.05 days**	**07 Dec '17**	**01 Jun '18**
135	**5165 Level**	**71.74 days**	**11 Jan '18**	**23 Mar '18**
152	**5150 Level**	**57.57 days**	**12 Feb '18**	**10 Apr '18**
172	**Ore Production**	**287.16 days**	**08 Jan '18**	**22 Oct '18**
173	**Surface**	**32.14 days**	**20 Sep '18**	**22 Oct '18**
183	**5225 Level**	**31.9 days**	**14 Sep '18**	**16 Oct '18**
189	**5210 Level**	**66.7 days**	**08 Jan '18**	**16 Mar '18**
222	**5195 Level**	**228.22 days**	**12 Feb '18**	**28 Sep '18**
256	**5180 Level**	**64.78 days**	**01 Jun '18**	**05 Aug '18**
283	**5165 Level**	**59 days**	**10 Apr '18**	**08 Jun '18**
297	**5150 Level**	**35.8 days**	**06 Apr '18**	**12 May '18**
332	**Teardown - Demob - Closure**	**30 days**	**22 Oct '18**	**21 Nov '18**



Project: Fenelon Project and LOMP
Date: 22 Dec '16

Task | Split | Milestone | Summary | Project Summary | Deadline | Progress

APPENDIX VIII – FINANCIAL MODEL

CASH-FLOW PROJECTION - FENELON 400 TONNES PER DAY , CAMFLO MILL, DUMAS U/G, GALARNEAU TRANSPORTATION, ATCO CAMP, OUTLAND CATERING

Dates for NPV calculation				30-Sep-16	31-Dec-16	31-Mar-17	30-Jun-17	30-Sep-17	31-Dec-17	31-Mar-18	30-Jun-18	30-Sep-18	31-Dec-18		
	Units	Rate	Freq	Q3 2016	Q4 2016	Q1 2017	Q2 2017	Q3 2017	Q4 2017	Q1 2018	Q2 2018	Q3 2018	Q4 2018	Total	PER TONNE
TOTAL DILUTED ORE TONNES									1789	30997	29513	29208	5216	96 721	
TOTAL DILUTED MINED GRADE GPT									5.49	8.94	10.68	8.95	6.93	9.3	
TOTAL MINED AU GRAMS									9823	277088	315083	261447	36129	899570	
TOTAL MINED AU OUNCE									316	8909	10130	8406	1162	28922	
Mill Recovery X payability		97.00%							97%	97%	97%	97%	97%	97%	
Recovered Troy ounces									306	8 641	9 826	8 154	1 127	**28 054**	
Tonnes / oz of recovered Au									5.84	3.59	3.00	3.58	4.63	3.45	
Price of gold - $US	**1 285.28**														
Exchange Rate	**1.31**														
Price of gold - $Cdn	**1 689.41**														
TOTAL REVENUE				-	-	-	-	-	**517 544**	**14 598 905**	**16 600 758**	**13 774 854**	**1 903 523**	**47 395 584**	$ **490.02**
PRE-PRODUCTION														-	
PERMITS/APPROVALS	Lump Sum	200 000	1			100 000	100 000							200 000	$ **2.07**
ENGINEERING	Lump Sum	400 000	1			200 000	120 000	80 000						400 000	$ **4.14**
CLOSURE COSTS	Lump Sum	896 217	1				448 109						448 109	896 217	$ **9.27**
PRE-PRODUCTION				-	-	**300 000**	**668 109**	**80 000**	**0**	**0**	**0**	**0**	**448 109**	**1 496 217**	$ **15.47**
CAPITAL															
MINING CONTRACTOR MOBILIZATION	LUMP SUM	$ 411 340.00	1				102 835	308 505						411 340	$ **4.25**
SITE SETUP	LUMP SUM	$ 740 128.00	1					681 264	58 864					740 128	$ **7.65**
POLISHING POND	Lump Sum	$ 363 858	1					363 858						363 858	$ **3.76**
MINE ACCESS ROAD UPGRADE	Kilometer	$ 50 000	5.00					125 000	125 000					250 000	$ **2.58**
DEWATERING - PIT	LUMP SUM	$ 182 500.00	1					109 500	73 000					182 500	$ **1.89**
UNDERGROUND WASTE DEVELOPMENT (RAMP)	Meters	$ 2 963	491						654 941	800 483				1 455 424	$ **15.05**
UNDERGROUND WASTE DEVELOPMENT (LEVEL ACCESS)	Meters	$ 2 467	484						477 118	715 677				1 192 795	$ **12.33**
REHABILITATE EXISTING WORKINGS	Lump Sum	$ 377 100	1						377 100					377 100	$ **3.90**
VENTILATION RAISES AND ESCAPEWAY	Meters	$ 2 627	101							212 234	53 059			265 293	$ **2.74**
CAPITAL COSTS				-	-	**0**	**102 835**	**1 588 127**	**1 766 023**	**1 728 394**	**53 059**	**0**	**0**	**5 238 437**	$ **54.16**
OPERATING															
UNDERGROUND ORE DEVELOPMENT	Tonnes	$ 32.75	81935						53 673	858 765	831 928	805 092	134 182	2 683 640	$ **27.75**
BACKFILL STOPES	Tonnes	$ 15.02	61346							128 970	267 153	497 457	27 637	921 217	$ **9.52**
CONTRACTOR INDIRECT COSTS	LUMP SUM	$ 10 563 600.00	1				104 580	1 119 742	2 330 330	3 632 822	1 816 939	1 393 339	165 848	10 563 600	$ **109.22**
SITE TEAR DOWN	LUMP SUM	$ -	1										0	-	$ **-**
SITE DEMOBILIZATION	LUMP SUM	$ -	1										0	-	$ **-**
CRUSHING	Tonnes	$ 4.87	96721						9 418	150 692	145 983	141 274	23 546	470 912	$ **4.87**
TRANSPORTATION TO MILL (use longest route)	Tonnes	$ 34.13	96720						66 021	1 056 337	1 023 327	990 316	165 053	3 301 054	$ **34.13**
DEWATERING - UNDERGROUND	LUMP SUM	$ 112 568.00	1						56 284	56 284				112 568	$ **1.16**
UNDERGROUND ROCK DEVELOPMENT (SILLS)	Meters	$ 1 924	724						348 340	557 344	487 676			1 393 361	$ **14.41**
VENTILATION	LUMP SUM	$ 306 303.00	1						122 521	137 836	45 945			306 303	$ **3.17**
MILLING	Tonnes	$ 37.00	96720						71 573	1 145 165	1 109 378	1 073 592	178 932	3 578 640	$ **37.00**
DELINEATION DRILLING	Meters	$ 109	3000						65 400	98 100	98 100	65 400		327 000	$ **3.38**
REFINING (0.11% OF TOTAL REVENUE)	% of revenue	0.11%	47395584						569	16 059	18 261	15 152	2 094	52 135	$ **0.54**
OPERATING COSTS				-	-	-	**104 580**	**1 119 742**	**3 124 130**	**7 838 374**	**5 844 691**	**4 981 622**	**697 291**	**23 710 429**	$ **245.14**
REMOTE CAMP OPERATION															
SALBAIE ROAD MAINTENANCE (90 KM) 2016 budget	Kilometer	$ 2 718	90					24 459	12 230	73 378	73 378	48 919	12 230	244 593	$ **2.53**
ROAD & SITE MAINTENANCE (18 KM)	Monthly	$ 11 614	18					20 905	10 453	62 716	62 716	41 810	10 453	209 052	$ **2.16**
CAMP SETUP/DEMOB	Lump Sum	$ 508 900	1					339 267					169 633	508 900	$ **5.26**
CAMP MONTHLY FEE	Monthly	$ 9 988	114					189 770	189 770	189 770	189 770	189 770	189 770	1 138 620	$ **11.77**
CAMP CATERING & JANITORIAL	Manhour	$ 59.00	15 930					103 386	187 974	211 471	211 471	169 177	56 392	939 870	$ **9.72**
REMOTE CAMP OPERATION				-	-	-	**0**	**677 787**	**400 426**	**537 334**	**537 334**	**449 676**	**438 478**	**3 041 035**	$ **31.44**
GENERAL & ADMINISTRATIVE															
OWNER'S COSTS	Days	$ 6 320	425					268 600	402 900	537 200	671 500	537 200	268 600	2 686 000	$ **27.77**
FIRST NATION PARTICIPATION	Percent	$ 1	-				-	-	-	-	-			-	$ **-**
FIRST NATION RECURRING COSTS	Monthly	$ 10 000	18					30 000	30 000	30 000	30 000	30 000	30 000	180 000	$ **1.86**
GENERAL & ADMINISTRATIVE				-	-	-	-	**298 600**	**432 900**	**567 200**	**701 500**	**567 200**	**298 600**	**2 866 000**	$ **29.63**
CONTINGENCY															
CONTINGENCY	Lump Sum	$ 3 615 850	1			28 927	65 085	376 048	560 457	1 048 597	705 091	592 999	238 646	3 615 850	$ **37.38**
CONTINGENCY				-	-	**28 927**	**65 085**	**376 048**	**560 457**	**1 048 597**	**705 091**	**592 999**	**238 646**	**3 615 850**	$ **37.38**
TOTAL OPERATING COSTS				-	-	**328 926.80**	**940 608.44**	**4 140 303.44**	**6 283 935.37**	**11 719 899.55**	**7 841 674.50**	**6 591 497.18**	**2 121 122.87**	**39 967 968**	$ **413.23**
ROYALTIES															
ROYALTY TO BALMORAL		**1%**	**NSR**	-	-	-	-	-	3 794	123 813	144 498	116 958	15 574	404 638	$ **4.18**
ROYALTY TO CYPRUS		**1%**	**NSR**	-	-	-	-	-	3 794	123 813	144 498	116 958	15 574	404 638	$ **4.18**
TOTAL ROYALTIES				-	-	-	-	-	**7 588**	**247 627**	**288 996**	**233 916**	**31 149**	**809 275**	$ **8.37**
ALL-IN-SUSTAINING COSTS - PRE TAX				-	-	328 927	940 608	4 140 303	6 291 523	11 967 526	8 130 670	6 825 413	2 152 272	40 777 243	
				-	-	(328 927)	(940 608)	(4 140 303)	(5 773 979)	2 631 379	8 470 088	6 949 441	(248 748)		
CUMULATIVE CASHFLOW - PRE TAX				-	-	**(328 927)**	**(1 269 535)**	**(5 409 839)**	**(11 183 818)**	**(8 552 440)**	**(82 352)**	**6 867 089**	**6 618 340**	**6 618 340**	$ **68.43**
DISCOUNT FACTOR		**5%**													
Net Present Value - Pre Tax (NPV)		**$5 841 520**													
Internal Rate of Return - Pre Tax (IRR)		**92%**													
ALL-IN-SUSTAINING COSTS PER TONNE PRE TAX, CAD				$ -	$ -	$ -	$ -	$ -	$ **20 537.38**	$ **1 384.91**	$ **827.44**	$ **837.10**	$ **1 910.18**	$ **1 453.50**	
ALL-IN-SUSTAINING COSTS PER TONNE PRE TAX, USD				$ -	$ -	$ -	$ -	$ -	$ **15 624.52**	$ **1 053.61**	$ **629.50**	$ **636.85**	$ **1 453.24**	$ **1 105.80**	
TAXES															
TAX QC, CAN, Duties									(1 096 783)				4 462 439	3 365 656	$ **34.80**
TOTAL TAXES				-	-	-	-	-	**(1 096 783)**	-	-	-	**4 462 439**	**3 365 656**	$ **34.80**
ALL-IN-SUSTAINING COSTS - POST TAX				-	-	328 927	940 608	4 140 303	5 194 740	11 967 526	8 130 670	6 825 413	6 614 711	44 142 899	
				-	-	(328 927)	(940 608)	(4 140 303)	(4 677 196)	2 631 379	8 470 088	6 949 441	(4 711 187)		
CUMULATIVE CASHFLOW - POST TAX				-	-	**(328 927)**	**(1 269 535)**	**(5 409 839)**	**(10 087 035)**	**(7 455 657)**	**1 014 431**	**7 963 872**	**3 252 684**	**3 252 684**	$ **33.63**
DISCOUNT FACTOR		**5%**													
Net Present Value - Post Tax (NPV)		**$2 802 166**													
Internal Rate of Return - Post Tax (IRR)		**60%**													
ALL-IN-SUSTAINING COSTS PER TONNE POST TAX, CAD				$ -	$ -	$ -	$ -	$ -	$ **16 957.16**	$ **1 384.91**	$ **827.44**	$ **837.10**	$ **5 870.68**	$ **1 573.47**	
ALL-IN-SUSTAINING COSTS PER TONNE POST, USD				$ -	$ -	$ -	$ -	$ -	$ **12 900.74**	$ **1 053.61**	$ **629.50**	$ **636.85**	$ **4 466.33**	$ **1 197.07**	