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NI 43-101 Technical Report & Preliminary Economic Assessment for Sleeping Giant Project, Eeyou Istchee James Bay, Québec, Canada

Prepared for

Abcourt Mines Inc.
475 Church Avenue
Rouyn-Noranda
(Evain district), Quebec, J0Z 1Y1

Project Location
Latitude: 49°08' North; Longitude: 77°58' West
Province of Quebec, Canada

Prepared by:
Olivier Vadnais-Leblanc, P.Geo.
Éric Lecomte, P.Eng.
InnovExplo Inc.
Val-d'Or (Québec)

Effective Date: June 7, 2023
Signature Date: July 19, 2023

SIGNATURE PAGE – INNOVEXPLO

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Olivier Vadnais-Leblanc, P.Geo.
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NI 43-101 Technical Report & Preliminary Economic Assessment for Sleeping Giant Project, Eeyou Istchee James Bay, Québec, Canada

Prepared for

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Effective Date: June 7, 2023

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Luc Boutin, P.Eng.
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Signed at Amos on July 19, 2023

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NI 43-101 Technical Report & Preliminary Economic Assessment for Sleeping Giant Project, Eeyou Istchee James Bay, Québec, Canada

Prepared for

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Project Location
Latitude: 49°08' North; Longitude: 77°58' West
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Signed at North Tetagouche on July 19, 2023

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NI 43-101 Technical Report & Preliminary Economic Assessment for Sleeping Giant Project, Eeyou Istchee James Bay, Québec, Canada

Prepared for

Abcourt Mines Inc.
475 Church Avenue
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(Evain district), Quebec, J0Z 1Y1

Project Location
Latitude: 49°08' North; Longitude: 77°58' West
Province of Quebec, Canada

Effective Date: June 7, 2023

(Original signed and sealed)

Marc L'Écuyer, P. Eng.
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Montréal (Québec)

Signed at Val D'or on July 19, 2023

(Original signed and sealed)

Jacques Blanchet, P.Eng. M.Sc.
Englobe Corp.
Montréal (Québec)

Signed at Quebec on July 19, 2023

CERTIFICATE OF AUTHOR – OLIVIER VADNAIS-LEBLANC

I, Olivier Vadnais-Leblanc, P.Geo. (OGQ No. 1082), do hereby certify that:

1. I am a professional geoscientist working for InnovExplo Inc., located at 560 3e Avenue, Val-d'Or, Quebec, Canada, J9P 1S4.
2. This certificate applies to the report entitled "Preliminary Economic Assessment for the Sleeping Giant Project, Eeyou Istchee James Bay, Québec, Canada" (the "Technical Report") with an effective date of June 07, 2023, and a signature date of July 19, 2023. The Technical Report was prepared for Abcourt Mines Inc. (the "issuer").
3. I graduated with a bachelor's degree in Geology (B.Sc.) from Université du Quebec à Montréal (Montreal, Quebec) in 2006.
4. I am a member in good standing of the Ordre des Geologues du Quebec (OGQ, No. 1082).
5. My relevant experience includes a total of 16 years since graduating from university. I acquired my mining expertise in the Goldcorp Eleonore mine and my exploration experience at Goldcorp's Eleonore project. I have been a consulting geologist for SGS from 2017 to 2022 and a consulting geologist for InnovExplo Inc. since February 2022.
6. I have read the definition of a qualified person ("QP") set out in National Instrument 43101/ Regulation 43101 ("NI 43101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43101) and past relevant work experience, I fulfill the requirements to be a QP for the purposes of NI 43101.
7. I am author and responsible for the preparation of items 4 to 12, 14.1.1 to 14.1.10, 14.1.12 to 14.1.13, 14.2.1, 14.3 and 23. I am also co-author and share responsibility for the relevant portions of items 1, 2, 3, 25, 26 and 27.
8. I have not visited the Property for the purpose of the Technical Report.
9. I have not had any prior involvement with the property that is the subject of the Technical Report.
10. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
11. I am independent of the issuer applying all of the tests in section 1.5 of NI 43101.
12. I have read NI 43 101 and Form 43 101F1, and the items of the Report for which I was responsible have been prepared in accordance with that instrument and form.

Signed this 19th of July 2023 in Montreal, Quebec.

(Original signed and sealed)

Olivier Vadnais-Leblanc, P.Geo. (OGQ No. 1082)
InnovExplo Inc.
olivier.vadnais-leblanc@innovexplo.com

CERTIFICATE OF AUTHOR – ERIC LECOMPTE

I, Eric Lecomte, P.Eng. (OIQ No. 122047, EGBC, No. 238548, PEO, No. 100574333), do hereby certify that:

1. I am a Senior Engineer working for InnovExplo Inc., located at 560 3e Avenue, Val-d'Or, Quebec, Canada, J9P 1S4.
2. This certificate applies to the report entitled "NI 43-101 Technical Report & Preliminary Economic Assessment for Sleeping Giant Project, Eeyou Istchee James Bay, Québec, Canada" (the "Technical Report") with an effective date of June 07, 2023, and a signature date of July 19, 2023. The Technical Report was prepared for Abcourt Mines Inc. (the "issuer").
3. I graduated with a bachelor's degree in Mining Engineering (B.Sc.A.) from Université Laval (Quebec City, Quebec) in 1998.
4. I am a member of the Ordre des Ingénieurs du Quebec (OIQ, No. 122047), the Association of Professional Engineers and Geoscientists of British Columbia (EGBC, No. 238548) and the Association of Professional Engineers of Ontario (PEO, No. 100574333).
5. I have worked as a mining engineer for a total of twenty-one (21) years since graduating from university. My expertise was acquired while working as a mining engineer. During these years, I occupied different technical and operational positions related to mining engineering in underground and open-pit operations.
6. I have read the definition of a qualified person ("QP") set out in National Instrument 43101/ Regulation 43101 ("NI 43101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43101) and past relevant work experience, I fulfill the requirements to be a QP for the purposes of NI 43101.
7. I am author and responsible for the preparation of items 14.1.11, 14.2.2, 16, 19, 22 and 24. I am also co-author and share responsibility for the relevant portions of items 1, 2, 3, 21, 25, 26 and 27.
8. I have visited the Sleeping Giant property that is the subject of the Technical Report, on February 12, 2023.
9. I was involved with the operation of the Sleeping Giant mine as a mine supervisor from 2002 to 2003. At the time, the property, that is the subject of the Technical Report, belonged to Cambior, now Iamgold.
10. I am independent of the issuer applying all the tests in section 1.5 of NI 43101.
11. I have read NI 43101, and the items of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
12. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed this 19th day of June 2023 in Val-d'Or, Canada.

(Original signed and sealed)

Eric Lecomte, P.Eng. (OIQ No. 122047)
InnovExplo Inc.
Eric.lecomte@Innovexplo.com

CERTIFICATE OF AUTHOR – LUC BOUTIN

I, Luc Boutin, P.Eng. (OIQ No. 112746, NAPEG No. L3892, PEO No. 100538487), do hereby certify that:

1. I am a professional engineer, employed as electrical project engineer for WSP Canada Inc
2. Located at 3 Rue Principale N, bureau 200, Amos, Quebec J9T 2K5.
3. This certificate applies to the report entitled "NI 43-101 Technical Report & Preliminary Economic Assessment for Sleeping Giant Project, Eeyou Istchee James Bay, Québec, Canada" (the "Technical Report") with an effective date of June 07, 2023 and a signature date of July 19, 2023. The Technical Report was prepared for Mine Abcourt Inc. (the "Issuer").
4. I graduated with a bachelor's degree in electrical engineering from the Université du Québec à Chicoutimi in 1994.
5. I am a member in good standing of the Ordre des Ingénieurs du Québec (OIQ No. 112746), the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (NAPEG No. L3892) and the Professional Engineers of Ontario (PEO, No. 100538487)
6. I have practiced my profession continuously as an engineer for a total of 25 years since graduating from university during which time I have been involved in ore treatment plant construction projects, pulp and paper amelioration projects and maintenance and, wood transformation plant construction projects.
7. I have read the definition of "qualified person" set out in National Instrument/Regulation 43101 ("NI 43101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43101) and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of that instrument.
8. I visited on April 12th and 27th 2023 the property for the purpose of this Technical Report.
9. I am author and responsible for the preparation of items 18.1 to 18.14.1 and 18.17. I am also co author and share responsibility for the relevant portions of items 1, 2, 21, 25, 26 and 27.
10. I am independent of the Issuer in accordance with the application of section 1.5 of NI 43101.
11. I have no prior involvement with the property that is the subject of the Technical Report.
12. I have read NI 43101 and Form 43101F1 and the items of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.
13. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading. Signed this 19th day of July 2023 in Amos, Québec, Canada.

(Original signed and sealed)

Luc Boutin, P. Eng.
WSP Canada Inc.
luc.boutin@wsp.com

CERTIFICATE OF AUTHOR – GUY COMEAU

I, Guy Comeau, P.Eng., do hereby certify that:

1. I am a professional engineer, employed as Senior Metallurgist for Soutex Inc., located at 1990, rue Cyrille-Duquet Local 204, Québec, Quebec, Canada, G1N 4K8.
2. This certificate applies to the report entitled "NI 43-101 Technical Report & Preliminary Economic Assessment for Sleeping Giant Project, Eeyou Istchee James Bay, Québec, Canada " (the "Technical Report") with an effective date of June 07, 2023 and a signature date of July 19, 2023. The Technical Report was prepared for Mine Abcourt Inc. (the "Issuer").
3. I graduated with a bachelor's degree in Metallurgical Engineering Co-operative Program from Technical University of Nova Scotia in Halifax, Nova Scotia in 1990.
4. I am a member in good standing of the "Ordre des ingénieurs du Québec" (OIQ licence No. 106546).
5. I have practiced my profession continuously as an engineer for a total of 32 years since graduating from university during which time I have been involved in mineral processing projects for precious and base metal properties in Canada. I acquired my expertise with Noranda, Xstrata, and Glencore. I have been a consulting engineer for Soutex Inc. since September 2008.
6. I have read the definition of "qualified person" set out in National Instrument/Regulation 43101 ("NI 43101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43101) and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of that instrument.
7. I have visited on February 7, 2023 the property for the purpose of this Technical Report.
8. I am author and responsible for the preparation of items 13 and 17. I am also coauthor and share responsibility for the relevant portions of items 1, 2, 3, 21, 25 and 26.
9. I am independent of the Issuer in accordance with the application of section 1.5 of NI 43101.
10. I have no prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43101 and Form 43101F1 and the items of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.
12. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading. Report not misleading.

Signed this 19th day of July 2023 in North Tetagouche, New Brusnwick, Canada.

(Original signed and sealed)

Guy Comeau, P.Eng.

CERTIFICATE OF AUTHOR – MARC L'ÉCUYER

I, Marc L'Écuyer, P. Eng. (OIQ No. 45289), do hereby certify that:

1. I am a professional engineer, employed as Project Director in the Geosciences division for Englobe Corp., located at 1200, boul. Saint-Martin Ouest, Laval, Canada, H7S 2E4.
2. This certificate applies to the report entitled "NI 43-101 Technical Report & Preliminary Economic Assessment for Sleeping Giant Project, Eeyou Istchee James Bay, Québec, Canada" (the "Technical Report") with an effective date of June 07, 2023 and a signature date of July 19, 2023. The Technical Report was prepared for Mine Abcourt Inc. (the "Issuer").
3. I graduated with a bachelor's degree in Mining Engineering from École Polytechnique de Montréal (Montreal, Quebec) in 1988.
4. I am a member in good standing of the Ordre des Ingénieurs du Québec (OIQ licence No. 45289).
5. I have practiced my profession continuously as an engineer for a total of 33 years since graduating from university during which time I have been involved from time to time in mining projects for precious and base metal properties in Canada. I acquired my expertise with Noranda Minerals inc. I have been a consulting engineer for Englobe Corp. since June 2021.
6. I have read the definition of "qualified person" set out in National Instrument/Regulation 43101 ("NI 43101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43101) and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of that instrument.
7. I previously visited the Sleeping Giant property, which is the subject of this technical report, as part of another mandate from April 18 to 20, 2022.
8. I am author for items 18.15, 18.16, 20 and co-author of and share responsibility for the relevant portions of items 1, 2, 21, 25 to 27.
9. I am independent of the Issuer in accordance with the application of section 1.5 of NI 43101.
10. I have no prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43101 and Form 43101F1 and the items of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.
12. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed this 19th day of July 2023 in Val-d'Or, Québec, Canada.

(Original signed and sealed)

Marc L'Écuyer, P. Eng.
Englobe Corp.
marc.lecuyer@englobecorp.com

CERTIFICATE OF AUTHOR – JACQUES BLANCHET

I, Jacques Blanchet, P.Eng. M.Sc. (OIQ No. 106232), do hereby certify that:

1. I am a professional engineer, employed as Senior Engineer for Englobe Corp. located at 1001, Sherbrooke Est Street, office 600, Montréal (Québec) H2L 1L3.
2. This certificate applies to the report entitled "NI 43-101 Technical Report & Preliminary Economic Assessment for Sleeping Giant Project, Eeyou Istchee James Bay, Québec, Canada." (the "Technical Report") with an effective date of June 07, 2023 and a signature date of July 19, 2023. The Technical Report was prepared for Mine Abcourt Inc. (the "Issuer").
3. I graduated with a bachelor's degree in Geological Engineering from Université Laval in Québec (Québec, Québec) in 1991.
4. I am a member in good standing of the Ordre des Ingénieurs du Québec (OIQ licence No. 106232).
5. I have practiced my profession continuously as an engineer for a total of 30 years since graduating from university during which time I have been involved in environmental and water management and treatment projects. I have been a consulting engineer for Englobe Corp. since 1993.
6. I have read the definition of "qualified person" set out in National Instrument/Regulation 43101 ("NI 43101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43101) and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of that instrument.
7. I have not visited the property for the purpose of this Technical Report.
8. I am author for items 18.14.2, 18.14.3 and 18.18 to 18.20. I am co-author of and share responsibility for for the relevant portions of items 1, 2, 21.1 and 25 to 27.
9. I am independent of the Issuer in accordance with the application of section 1.5 of NI 43101.
10. I have no prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43101 and Form 43101F1 and the items of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.
12. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed this 19th day of July 2023 in Québec, Québec, Canada.

(Original signed and sealed)

Jacques Blanchet, P. Eng. M.Sc..
Englobe Corp.
jacques.blanchet@englobecorp.com

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1. SUMMARY

This Technical Report Preliminary Economic Assessment ("PEA") for the Sleeping Giant Project (the "Project") (the "Report") was prepared and compiled by InnovExplo Inc ("InnovExplo") at the request of Abcourt Mining Group ("Abcourt"). The Project is located 80 km north of the town of Amos in northwestern Quebec. The purpose of the Report is to summarize the results of the PEA for the Project, in accordance with the guidelines of the Canadian Securities Administrators National Instrument 43 101 ("NI 43-101") and Form 43101.

This Report was prepared based on contributions from several independent consulting firms including, InnovExplo Inc ("InnovExplo"), WSP Inc ("WSP"), Soutex Inc ("Soutex") and Englobe Corp ("Englobe"). It provides a base case assessment for developing the Sleeping Giant gold deposit as an underground mine, continuing from the historic mine, including all relevant facilities.

All monetary units in the Report are in Canadian dollars ("CAD" or "$"), unless otherwise specified. Costs are based on first quarter (Q1) 2023 dollars. Quantity and grades are rounded to reflect that the reported values represent approximations.

1.1 Contributors

The major PEA contributors and their respective areas of responsibility are presented Table 1.1. Each contributor is in good standing with their respective professional associations and is considered a Qualified Person (QP) as defined in National Intstrument 43-101. The contributors' certificates specify the sections in the Report for which each QP is responsible. Each QP has also contributed figures, tables, and parts of Chapter 1 (Summary), Chapter 2 (Introduction), Chapter 25 (Interpretation and Conclusions), Chapter 26 (Recommendations), and Chapter 27 (References).

Table 1.1 – Report contributors

Qualified Person	Company	Areas of report responsibility
Eric Lecomte	InnovExplo Inc ("InnovExplo")	Items 14.1.11, 14.2.2, 16, 19, 22 and 24 & Contributor for items 1, 2, 3, 21, 25, 26 and 27.
Olivier Vadnais-Leblanc	InnovExplo Inc ("InnovExplo")	Items 4 to 12, 14.1.1 to 14.1.10, 14.1.12 to 14.1.13, 14.2.1, 14.3 and 23 & Contributor for items 1, 2, 3, 25, 26 and 27.
Luc Boutin	WSP Inc ("WSP")	Items 18.1 to 18.14.1 and 18.17 & Contributor for items 1, 2, 21, 25, 26 and 27.
Guy Comeau	Soutex Inc ("Soutex")	Items 13 and 17 & Contributor for items 1, 2, 3, 21, 25 and 26.
Jacques Blanchet	Englobe Corp ("Englobe")	Items 18.14.2, 18.14.3 and 18.18 to 18.20 & Contributor for items 1, 2, 21.1 and 25 to 27.
Marc L'Écuyer	Englobe Corp ("Englobe")	Items 18.15, 18.16 and 20 & Contributor for items 1, 2, 21 and 25 to 27.

1.2 Property Description and Ownership

Abcourt's Sleeping Giant property (the "Property" or the "Project") is located 80 km north of the town of Amos (pop. 12,800) in northwestern Quebec. It is accessed from provincial highway Route 109, which runs through the Property and connects Amos to Matagami (pop. 1,450). A 1-km gravel road leads westward from the paved highway to the mine, and a network of forestry roads leads to other parts of the Property (Figure 5.1).

1.3 Mining Titles and Claim Status

The Property consists of a contiguous block of 132 claims converted into map-designated cells, and four (4) mining leases, collectively covering an area of 6480.4 ha (Figure 4.2) in the townships of Maizerets, Glandelet, Soissons and Chaste on NTS map sheets 32E01 and 32F04. All the mining titles are registered 100% to Abcourt. A detailed list of the land holding comprising the Property is provided in Appendix I.

Abcourt is the sole owner of all leases and mining titles comprising the Property.

1.4 Geological Setting and Mineralization

The Sleeping Giant deposit is hosted within the so-called first volcanic cycle of the Northern Volcanic Zone of the Archean Abitibi Subprovince (Chown et al., 1992). The first volcanic cycle corresponds to an extensive subaqueous basalt plain with scattered felsic volcanic edifices, interstratified with or overlain by volcaniclastic assemblages. The 1 to 3 km thick basaltic assemblage (Chown et al., 1992) is dominantly composed of tholeiitic massive, pillowed and brecciated volcanic rocks (Picard and Piboule, 1986), with minor chert, iron formation and volcaniclastic rocks (Mueller and Donaldson, 1992; Pilote, 1989). U-Pb dating of felsic centres in the middle to upper part of the first volcanic cycle indicates a time interval of 2730-2720 Ma (Mortensen, 1993). The Northern Volcanic Zone is interpreted as a coherent assemblage, initially formed from a diffuse and immature arc (Chown et al.,1992).

The Northern Volcanic Zone was affected by a N-S shortening event from 2708 to 2685 Ma; this comprised a succession of several tectonic pulses of continuous deformation (Chown et al., 1992). The main deformational features are: (1) E-W trending, subvertical, regional folds with an axial-planar fabric; (2) major E-W trending, 1 to 4 km wide reverse deformation zones of regional extent; and (3) dextral, 1 to 5 km wide, NW-SE trending deformation zones.

At the district scale, three voluminous synvolcanic and polyphase (diorite-tonalite leuco-tonalite) plutons (Chown et al., 1992) are responsible for the disturbed regional structural trend (showing the location of the Sleeping Giant mine close to the central part of a triple junction of structural trends. Map modified from Hocq (1990) and Chown et al. (1992).

Figure 7.3). The Sleeping Giant mine is located close to the centre of a triple junction zone in the structural trend (Daigneault and Archambault, 1990). Other important features of the vicinity of the Property include the E-W trending Laflamme fault and the NW-SE trending Hanicana fault.

The volcanic tectonostratigraphy in the area defines two volcanic cycles (North Cycle and Mine Cycle) in relation to a large intrusive complex. At the base of the stratigraphic sequence is the Northern Cycle (northwest part of the property), which contains mainly iron-rich tholeiitic basalts and co-magmatic sills of gabbro. These tholeiites are easily distinguished from the tholeiites of the Mine Cycle by their higher TiO_2 content (>1, 2%). Stratigraphically above and concordant with the Northern Cycle, the Mine Cycle sequence represents the dominant host sequence of the Sleeping Giant deposit. This cycle contains mainly magnesium-rich tholeiitic basalts and co-magmatic gabbro sills. Laminar deposits composed of fine sediments, tuffs and iron formations (with magnetite) are inter-stratified in the sequence. These sedimentary and volcaniclastic rocks define larger amalgamated units in the central part of the Mine Cycle.

The stratigraphic sequence of the Mine Cycle is intersected by intrusions of intermediate to felsic composition and calc-alkaline affinity that constitute the Sleeping Giant Complex and are penecontemporaneous with the Mine Cycle volcanism. The Sleeping Giant Complex includes a main dacitic mass, several satellite dacitic units and a swarm of felsic porphyry dykes. Four main phases are recognized in the magmatic evolution of the complex:

1. Dacite with mafic phenocrysts (chlorite spots);
2. Dacite with feldspar phenocrysts and feldspar;
3. Porphyry with quartz+feldspar phenocrysts (locally with granitoid texture);
4. Porphyry with quartz phenocrysts.

The mine sequence is represented by a volcano-sedimentary succession which is intruded by a felsic complex and transected by late hornblende-rich dykes. Strata strike predominantly E-W and dip steeply to the south, forming a single homoclinal stratigraphic succession. All the rock types have been metamorphosed to greenschist facies.

Gold-bearing sulphide-quartz veins (the most economically important) are generally massive, ranging from a few centimetres to 2 metres thick (averaging 50 cm for the mined veins). The veins are gold-rich, with assays commonly exceeding 100 g/t Au (unpublished. data, Cambior Inc.). The vein contacts range from sharp, planar and free of surrounding planar fabrics to wavy and schistose. Branching of the main veins is a common phenomenon but does not show consistent attitudes. Veins terminate laterally through pinching out or arborescent multi-branching veinlets.

The deformation affecting the host rocks is expressed by: (1) the subvertical attitude of the volcanic strata; (2) the development of ductile planar and linear elements; (3) local mesoscopic folds; and (4) subhorizontal extensional calcite veinlets. These features are related to the regional deformation event. Subsequent deformation includes local shear zones and late brittle faults.

1.5 Deposit Type

Sleeping Giant is a sulphide-rich lode gold deposit of volcanogenic affinity. In their geochemistry study of the VMS deposits in the Abitibi belt, Gaboury and Pearson (2008) classified the Sleeping Giant Rhyolite as “F1 type”, which includes VMS deposits rich in gold and silver. Such deposits, which have the particularity of being spatially isolated, are probably the result of local hydrothermal processes (Gaboury, 2004). The Sleeping Giant deposit displays atypical characteristics of orogenic Archean deposits associated with major faults (Figure 7.8). According to Gaboury (1999), gold-bearing veins formed in subhorizontal strata shortly after QFP injections but before the end of mafic magmatism. All rock types, including gold-bearing veins, have been affected by regional ductile deformation (folding of strata and development of schistosity) and metamorphism to the greenschist facies.

1.6 Drilling

The Project is characterized by 1,185,868.63 m of drilling records (8,433 drill holes). These holes have been drilled since the early days of initial exploration on the Property, continuing through different production phases under various operators. Between 2020 and 2022, the Issuer completed a significant amount of underground exploration and definition drilling. The Issuer disclosed the previously unpublished results in a press release on November 10, 2022 (available on SEDAR). The results were from 94 drill holes totalling 9,281 m drilled underground from levels 235 and 295 between November 2020 and May 2022. Eighty-two (82) holes intersected at least one interval grading over 2.0 g/t Au. Of the 9,281 m drilled during this campaign, 770 m of core was sampled, yielding 1,140 assayed samples.

All samples were analyzed at the mine's internal laboratory. In the autumn of 2022, Abcourt sent 250 (22%) of the 1,140 samples to a certified and independent laboratory as part of a re-assay and quality control program to verify and validate the initial results from the internal laboratory. Most samples (243) were from mineralized intervals grading more than 2.0 g/t Au. 167 of these (69%) were sent to the external laboratory as pulps and coarse rejects. The results of the independent re-assays confirmed the validity of the internal laboratory results.

1.7 Mineral Resource Estimate

The mandate assigned to InnovExplo was to create a 3D interpretation of the mineralized veins of the Sleeping Giant deposit and to update the resources using the holes drilled since 2019 (the "2022 MRE"). This is the first 3D model built for this deposit. The model contains 846 veins.

Mineralized zones in the deposit are narrow, with an interpreted average thickness of 0.7 m. The real in-situ thickness of the veins is often less than 0.7 m, but the assay length is rarely shorter than 0.5 m (Figure 14.4). Veins are built with at least one (1) assay. The general minimum assay length is 0.5 m. Of the 288,388 assays, the vast majority (286,678) were more than 0.5 m long. The minimum modelling grade is 2 g/t Au over 0.5 m. A margin of 10 m was set around the most external drill hole intercept. If a drill hole not selected for the interpreted zone is located in the margin area, the margin is automatically set at half the distance between the holes. The 3D modelling was done using Genesis software.

The gold assays were composited at 0.5 m within all defined mineralized zones to minimize any bias introduced by variable sample lengths.

Most sampled intervals from mineralized zones are 0.5 m, and the average interval length is 0.7 m. Many zones are locally built on a single 0.5 m assay.

The drill hole dataset contains 39,851 composites with an average length of 0.48 m and a median length of 0.49 m. A total of 3,713 composites are based on assay intervals that are exactly 0.5 m. The smallest composites are 0.10 cm, and the longest are 0.67 m. All composites less than 0.1 cm long were redistributed among the other composites in the same interval (Figure 14.5). Compositing was done in Genesis using drill hole intervals that cut through veins.

Each mineralized zone solid was estimated separately using its own set of composites. A grade of 0 g/t Au was assigned to missing sample intervals.

The deposit is divided by an E-W trending brittle fault. Capping levels have been determined for veins north of and south of this fault. Results from both sides are similar, with a capping grade around 95 g/t Au. For consistency, a single capping grade of 95 g/t Au was used for all composites throughout the deposit, resulting in 484 capped composites out of 39,851. This reduced the contribution of the 1% highest-grade composites from 18.6% to 11.5% of contained gold, a reduction of 7.1%. If the capping grade had been determined using the method of 10 times the average composite grade, it would be 90 g/t Au.

Cambior measured the density factor of the historical ore (mineralized zone) in 2001 and 2002 by collecting and analyzing three samples per month from the material processed at the mill. The results varied between 2.8 g/cm^3 and 2.9 g/cm^3 for an average of 2.86 g/cm^3. Cambior used a density factor of 2.85 g/cm^3 starting in December 2002 for both resource and reserve estimates (Asselin, 2008). This density factor yielded an acceptable reconciliation between produced and calculated tonnages (Jourdain et al., 2011). For the 2022 MRE, InnovExplo used the same global density factor of 2.85 g/cm^3.

A grade model was interpolated using the 0.5m capped composites (95 g/t Au) from conventional assay grade data. The interpolation method retained for the final resource estimate was inverse distance square ("ID2") with capping of high-grade values. The ID2 method was preferred because this deposit includes many high-grade gold values that locally create high-grade pockets of gold (Figure 14.23). The ordinary kriging (OK) interpolation method tends to smooth the grade and therefore minimize the impact of these high-grade pockets, assigning the higher grades to other blocks.

To meet the criterion of "reasonable prospects for eventual economic extraction" and follow CIM Guidelines (2019), the potential stope shapes (including 'must-take blocks') were optimized using the Deswik Stope Optimizer (DSO) from the Deswik software. The block model was generated after completing the mineral resource estimate. This allowed for more flexibility during the optimization process, including sub-shapes and anneal parameters to ensure maximum resource conversion to DSO. The additional parameters for the optimization process are summarized in Item 14 of the Report.

The dominant system ensures that all resources are associated with one of the evaluated categories (indicated or inferred) for the DSO-based resource classification. The category of each DSO is dictated by the most prominent category (by volume) included in each solid, following the 50%+1 rule.

Table 1.2 – 2022 MRE for the Sleeping Giant Mine (effective as of December 12, 2022)

	Potential Long Hole (Cut off at 4.25 g/t Au)			Potential Room and Pillar (Cut off at 5.00 g/t Au)			Total		
	Tonnes	Grade (Au g/t)	Ounces AU (oz)	Tonnes	Grade (Au g/t)	Ounces AU (oz)	Tonnes	Grade (Au g/t)	Ounces AU (oz)
Indicated Resources	677,000	7.03	153,000	78,000	7.98	20,000	755,000	7.14	173,300
Inferred Resources	677,000	8.13	177,000	207,000	10.67	71,000	884,000	8.74	248,300

Notes to the 2022 MRE:

1. The independent and qualified persons for the 2022 MRE, as defined by NI 43-101, are Olivier Vadnais-Leblanc (P.Geo.) and Eric Lecomte (P.Eng.), both from InnovExplo Inc.
2. These mineral resources are not mineral reserves because they do not have demonstrated economic viability. The results are presented undiluted and are considered to have reasonable prospects of economic viability. The 2022 MRE follows CIM Definition Standards (2014) and CIM Guidelines (2019).
3. The estimate encompasses 846 mineralized zones modelled using a minimum geological width of 0.5 m in Genesis software.
4. A density value of 2.85 g/cm^3 (based on measurements and mine and mill reconciliation) was assigned to all mineralized zones.
5. High-grade capping supported by statistical analysis was established at 95 g/t Au for all mineralized zones and applied to the composite data. Composites (0.5 m) were calculated within the zones using the grade of the adjacent material when assayed or a value of zero when not assayed.
6. The “Reasonable Prospect of Eventual Economical Extraction” requirement is fulfilled using cut-off grades based on reasonable mining parameters, locally constrained within Deswik Stope Optimizer shapes using a minimum mining width of 1.7 m for both potential methods. It is reported at a rounded cut-off grade of 4.25 g/t Au using the long-hole (“LH”) method and 5.0 g/t Au using the room and pillar (“R&P”) method. The cut-off grades were calculated using the following parameters: mining cost = C$213.96/t (LH) to C$261.56/t (R&P); processing cost = C$35.10/t; G&A = C$22.09/t; gold price = US$1,650.00/oz and USD:CAD exchange rate = 1.30. The cut-off grades should be re-evaluated in light of future prevailing market conditions (metal prices, exchange rates, mining costs etc.).
7. The estimate was completed using a sub-block model in Surpac 2022. A 4m x 4m x 4m parent block size was used (1m x 1m x 1m sub-blocked). Grade interpolation was obtained by Inverse Distance Squared (ID2 using hard boundaries.
8. The mineral resource estimate is classified as Indicated and Inferred. The Inferred category is defined with a minimum of three (3) drill holes within the areas where the drill spacing is less than 75 m and shows reasonable geological and grade continuity. The Indicated mineral resource category is defined with a minimum of four (4) drill holes within the areas where the drill spacing is less than 30 m and shows reasonable geological and grade continuity.
9. The number of metric tons was rounded to the nearest hundred, following the recommendations in NI 43-101, and any discrepancies in the totals are due to rounding effects. The metal contents are presented in troy ounces (tonnes x grade / 31.10348) rounded to the nearest hundred.
10. The independent and qualified persons for the 2022 MRE are not aware of any known environmental, permitting, legal, political, title-related, taxation, socio-political, or marketing issues that could materially affect the estimate.

1.8 Mining Method

Operations at the Sleeping Giant mine were suspended in 2014. The proposed plan would see a re-commencement of operations using a combination of long hole, room & pillars, and shrinkage mining methods. Underground work would begin with the rehabilitation of the shaft service compartment and the emergency raise, followed by the rehabilitation of the existing levels over the succeeding months. Mined material will be processed at a rate of approximately 350 tpd.

The planned underground operation will employ conventional mining methods, using track drift to haul mineralized material to the existing orepass system. To reach the optimized stopes, old rehabilitated drifts and new drifts are excavated from existing levels. Additionally, some new levels will be developed near the bottom of the shaft (below N-1060). The current operation has a remaining mine life close to six (6) years following the pre-production period.

The project will involve an underground operation with a mining method optimized to the deposit geometry (narrow veins) along with previously used production techniques. It will employ a variety of stoping methods (long hole, room & pillars, and shrinkage). Mining voids will not be backfilled, continuing the previous approach.

The optimization process has been completed using Deswik™ DSO (stope optimizer module), whereas the planification has been optimized using the Deswik™ Scheduler. Both programs ensure optimal and extensive outputs to be used in the final cashflow. Though not required at this phase of the project, this adds a significant value to the study and minimize risks associated with a simpler approach.

The two main objectives of the current assessment and schedule is to minimize the production ramp-up and to maximize resulting grade.

Parameters for the cut-off calculation are summarized in Table 1.3 for each method.

Table 1.3 – Cut-off grade parameters

Input Parameters	Unit	Long hole	Room & Pillars	Shrinkage
Gold Price	US$/oz	1,650	1,650	1,650
Exchange Rate	USD:CAD	0.77	0.77	0.77
Royalty	%	2.0	2.0	2.0
Refining Cost	$/oz	5.00	5.00	5.00
Processing Cost	$/t milled	35.10	35.10	35.10
Metallurgical Recovery	%	95.0	95.0	95.0
Mining Recovery	%	95.0	95.0	95.0
Mining Dilution	%	17.6	23.5	2.9
Mining Cost	$/t treated	214.35	261.94	249.08
General and Administration	$/t treated	21.70	21.70	21.70
Cut-Off Grade	**g/t**	**4.50**	**5.25**	**5.05**

The final resource with economic potential is presented in Table 1.4.

Table 1.4 – Economic mineralized material summary

Mining method	Indicated resources			Inferred resources		
	Tonne (t)	Grade (g/t)	Troy Ounces (oz)	Tonne (t)	Grade (g/t)	Troy Ounces (oz)
Long hole	57,000	6,08	11,100	140,200	7.69	34,700
Room & Pillars	161,600	6,19	32,200	147,400	9.01	42,700
Shrinkage	59,000	7,64	14,500	155,100	10.51	52,400
Total	277,600	6,48	57,800	442,700	9.12	129,800

1.9 Mineral Processing and Metallurgical Testing

The Sleeping Giant mine began his operation in 1988 and was last operational in 2014. A recovery model was developed based on historical data collected during processing of Sleeping Giant ore and is based on head grade.

According to historical daily reports, the processing plant was able to process on a constant basis between 700 and 750 tpd of ore.

1.10 Recovery Method

The Sleeping Giant mine was in operation from 1988 to 2014. From August 2016 until 2022, the processing plant was fed by ore from nearby mines. Ore is processed using two stages of crushing, rod mill grinding, ball mill grinding and a carbon-in-leach gold plant.

The expected gold recovery is based on a model that was developed from mill performances when processing. With a head grade of 8.1 g/t, the expected recovery is 96.7%.

The capacity of the processing plant is between 700 and 750 tpd and is higher than the current expected processing tonnage of 350 tpd on a 12 hours daily shift.

There are no planned modifications to the plant throughput, equipment size, number of equipment or ore that are expected to change this recovery model or the plant’s capacity.

1.11 Project Infrastructure

The project infrastructure includes:

- Site roads;
- Headframe;
- Ore processing plant;
- Surface main electrical sub-station;
- Administration building and dry complex;
- Maintenance Shop and Warehouse;
- Mine site entrance / guardhouse;
- First Aid / Emergency Services;
- Communication systems;
- Camp complex;
- Explosives plant and storage;
- Core shack;
- Fuel storage;
- Tailings pond.

The steel structure of the headframe is in good condition. Minor improvements will be required to the protective brattice to allow safe operations. The hoist building and the headframe silo building are fully functional and will not require any structural modifications. The headframe silo will require extensive rehabilitation and the waste rock silo chute will need to be rehabilitated. Both improvements will need to be completed prior to the start of operations.

The underground loading stations are showing signs of corrosion degradation and some structural elements will need to be rehabilitated prior to the start of production.

The processing plant does not require additional mechanical equipment to resume operations. Significant maintenance is required for the crushers, tanks and silos. Conveyors, pumps, crushers, and cranes will require normal maintenance to ensure their longevity. The building structure is in good condition.

The main 2.4 kV electrical substation will be updated for code compliance to “Use of electricity in mines”, increase reliability and safety for electrical personnel. The emergency generator, which is primarily used for mine dewatering, must be restored to working order. Communication systems are functional.

The dewatering system of the mine has always been maintained in operation and all levels are accessible; however, investments in piping will have to be made over time to ensure continuity of operations. No diesel equipment is planned underground, so no fuel distribution is required.

The surface infrastructure is in good condition. The existing buildings and access roads are well maintained and capable of supporting plant operations. Only minor improvements and repairs will be required to ensure long-term sustainability. This favorable infrastructure condition provides a solid foundation for the reopening of the mine and operations. To attract and retain labor a new camp complex will be built.

The camp complex will be made up of used prefabricated modular-type units and will include the following blocks: dormitories, cafeteria, and dining room. The drinking water provided by 2 groundwater wells is distributed in various buildings in the Sleeping Giant mine complex through a piping system. The design of a new wastewater treatment system will be required to accommodate the increased volume of wastewater to treat, for which a ministerial authorization from the Ministère de l’Environnement, de la Lutte contre les changements climatiques, de la Faune et des Parcs (MELCCFP) is mandatory.

Waste rock from the underground mine is stored in the waste rock pile located south of the headframe. The ore pad is located on the southwest side of the processing plant.

The current TSF is located north of the processing plant and is made up of three cells: Cell 1 (deposition pond), Cell 2 (recirculation pond), and Cell 3 (polishing pond). Dikes 1, 2, 3, 6, and 7 surround the TSF, while inner dike 1 and the median dike separate the three cells. An access road dividing Cell 2 was built in 2019. An emergency spillway is located at the western end of the TSF, in Cell 3.

For the Tailings Storage Facility (TSF), certain work will have to be carried out to be able to store the mine tailings generated by the 720,000 t of ore milled in the concentrator. It is proposed to split Cell 2 into two parts. This modification would allow the storage of the mine tailings in the future Cell 2A and the remaining Cell 2 would continue to be used for recirculation of mining process water to the processing plant. Cell 2A will be built by raising the dikes by lifts similar to those in Cell 1. The lifts of the various dikes will be built using tailings and waste rock. The anticipated tailings storage capacity of the future Cell 2A is estimated at 500,000 m^3.

Mining wastewater is generated primarily from the mining tailings. The mining wastewater treatment train is an annual batch process. The processing unit is located near Cell 3, which is also used as a treatment and polishing basin. On an annual basis, the treatment in the TSF begins in May-June and ends in October. The treatment of the mining wastewaters before their discharge into the environment will be carried out in a manner similar to that of the last years of operation of the site. For the resumption of operation of the Sleeping Giant mine, maintenance work or replacement is planned.

1.12 Environmental and Permitting

The Sleeping Giant mine was in production from 1988 to 2014. Since the temporary shutdown in 2014, dewatering operations have been maintained, the mine has been kept dry and all the infrastructures were kept operational. No addition or dismantling of infrastructures has occurred. The current mining plan of the Project consists of extracting 720,000 tons of ore, from new and legacy stopes that can be accessed almost exclusively from existing levels.

The Sleeping Giant mine is subject to Canadian and Québec requirements in term of monitoring studies and environmental controls. The mining site is subject to the federal metal and diamond mining effluent regulations.

The most recent annual report covers year 2021 and summarizes monitoring of effluents, groundwaters, emissions of atmospheric contaminants, waste production, and tailings management (GCM Consultants, 2022).

Environmental baseline studies have not been carried out but may be required. If necessary, these studies would cover the footprint of the new infrastructure.

1.13 Regulatory context and environmental studies

In Québec, the procedure for environmental assessment is included in the *Environment Quality Act* (EQA, Chapter Q-2) and its regulations. As the proposed Project remains below the thresholds mentioned in the EQA, it does not trigger the environmental impact assessment procedure. The Project is also below the thresholds of the federal legislation (*Impact Assessment Act and Physical Activities Regulations*) and as such, does not trigger a federal impact assessment study.

The proposed Project falls under the EQA and the *Regulation respecting the regulatory scheme applying to activities on the basis of their environmental impact* (Q-2, r. 17.1). The permitting process is triggered by the EQA Section 22.

The Project is also subject to a depollution attestation under Division III of the EQA. Technical advice received from the MELCCFP in March 2020 related to the depollution attestation mentioned that new monitoring wells to assess the groundwater quality is required, since some analytical results from improperly installed wells exceeded applicable criteria.

Throughout all stages of the Project (i.e., construction, operations, and closure) site – activities will be required to comply with provincial and federal acts and regulations. The detailed engineering and operations will consider the conditions, mitigation measures, and monitoring requirements associated with the authorization process.

The following are examples of permits that may be required by the provincial jurisdiction:

- *Mining Act* (M-13.1) and its associated regulation entitled Regulation respecting mineral substances other than petroleum, natural gas and brine (M 13.1, r. 2);
- *Environmental Quality Act* (Q-2) enacts more than 20 different regulations related to the protection of the environment and the living species inhabiting it;
- *Threatened or Vulnerable Species Act* (E-12.01);
- *Dam Safety Act* (S-3.1.01); and
- Directives and Guidelines, in particular the *Directive 019 sur l'industrie minière* (translated into *Directive 019 on the mining industry*; MDDEP, 2012).

The following are examples of permits, directives, and guidelines that may be required by the federal jurisdiction:

- Fisheries Act (R.S.C., 1985, c. F-14);
- Canadian Environmental Protection Act (S.C. 1999, c. 33);
- Species at Risk Act (S.C. 2002, c. 29); and
- Directives and Guidelines :
- Environment Canada Environmental Code of Practice for Metal Mines (EC, 2009);
- Guidelines for the Assessment of Alternatives for Mine Waste Disposal (EC, 2016); and
- Strategic climate change assessment (ECCC, 2020).

1.13.1 Considerations of social and community impacts

The Property is located in the southern part of the territory of the Eeyou Istchee James Bay Regional Government on Category III Lands. The Regional Government is governed under the laws of Québec and exercises the jurisdictions, functions, and powers over Category III Lands in the Eeyou Istchee James Bay Territory. The Eeyou Istchee James Bay Territory is made up of 16 communities inhabited by some 15,000 Crees and 15,000 Jamésiens.

The Regional Government is responsible for managing Category III Lands, which are public lands in the domain of the State. The Crees have exclusive trapping rights there (except in the southern zone), as well as certain non-exclusive hunting and fishing rights. Category III Lands include all of the lands within the territory covered by the James-Bay-Northern-Québec Agreement (JBNQA) that are located south of the 55th parallel and are not included in other land categories, notably Category I and II Lands for which Crees have exclusive hunting, fishing, and trapping rights. Further south, the Abitibiwinni First Nation of Pikogan is located about 75 km south of the mining site by road. This community is also present on the territory, carrying out hunting, fishing, and trapping activities.

Besides its energy potential, the area's economy is mainly oriented toward the exploitation of the natural resources, namely forestry and mining. Fishing, hunting, and trapping remain key activities, attracting tourism to outfitter, while being an integrated part of the culture and identity of the First Nations. For instance, Crees are well part of the economy and hold numerous companies in various domains such as construction and transportation.

So far, there has been no negotiation nor any agreement involving Abcourt and any group, organization, or First Nations regarding the Project.

Impacts on any components are triggered by a source of impacts, which could be project infrastructures, works, or activities. Impacts are typically observed during the construction, the operation, or the closure phase. Impacts are determined for each biophysical and social component, through the analysis of the technical characteristics of the Project, the knowledge of the surrounding environment, and experience from similar projects.

Considering the key features of the Project and the setting of the receiving environment, potential impacts of the project will be influenced by a few features that will help to avoid and minimize the overall impacts:

- The underground mine and its infrastructures are already in place;
- Ore processing will be performed on site, with the processing plant and most of the infrastructures also in place; and
- The mining site is located away from any minicipality or public/private infrastructure.

With the implementation of mitigation measures in the course of the Project, these features will contribute to avoiding and to minimizing the overall impacts of the project on the environmental and social components.

1.13.2 Mine reclamation and closure plan

The *Mining Act* (CQLR, c. M-13.1) and the *Regulation concerning mineral substances other than petroleum, natural gas and brine* (CQLR, c. M-13.1, r. 2) require companies to restore the land affected by their mining and mineral exploration activities (Section 232.1). The *Mining Act* requires that a rehabilitation and restoration plan (also called closure plan) and financial guarantee covering the total cost of restoration work be provided by corporations. The plan must be reviewed every five years. Significant changes to the Project may also trigger the need to update the plan, as requested by the MRNF.

In 2018, an updated version of the rehabilitation and restoration plan for the Sleeping Giant mine was filed to the MRNF by Abcourt. Subsequently, Abcourt received comments from the MRNF on the closure plan. Responses were submitted to the MRNF in February 2023 in which a revised cost estimate for site closure was included. Abcourt is currently waiting for further feedback from the MRNF.

The restoration concept approved by the MRNF in the previous versions of the closure plan is maintained. Agricultural lime used as an amendment will be mixed with the first 30 cm of mine tailings for the restoration of the TSF. It is anticipated that the geochemical characteristics of the anticipated ore feed, and therefore of future mine tailings, will be similar to the former mineralized zones previously mined. Additional tests will be required to assess the dosage required for the anticipated agricultural lime used as an amendment to meet the current requirements of the *Guide de caractérisation des résidus miniers et du minerai* (MELCC, 2020).

In addition to the environmental monitoring program, a physical stability monitoring program will be implemented following the site closure to evaluate the integrity of the existing structures. Following revegetation of the site, agronomic monitoring will be carried out. The monitoring programs will be conducted during the post-mining and post-restoration periods. The post-mining period is expected to be two years.

The cost estimate provided takes into consideration the dismantlement of surface structures, new accommodation facilities, water management infrastructure and the restoration of the TSF.

1.14 Capital and Operating Costs Estimates

1.14.1 Capital costs

The cumulative life of mine (LOM) capital expenditure ("CAPEX") costs, including initial investment capital cost (pre-production period) and the sustaining cost are estimated at $97.8 millions ($M). This estimate was prepared with an expected accuracy of +/- 30% of the final Project cost. The capital cost estimate was compiled using a mix of quotations and budgetary quotations, database costs and factors.

A summary of the capital cost and a cost profile over the life of mine are presented in Table 1.5 and in Figure 1.1

Table 1.5 – Project capital cost summary

Capital Expenses[1] (M CAD)	Total Capital Costs ($M)
Pre-production operating expenses	27,1
Mine development	28,7
Infrastructures	6,0
Equipment reconditioning	6,7
Electrical circuit	6,4
Tailing facilities	10,9
Closure and reclamation	12.1
Total capital cost	**97.8**
Pre-production revenue NSR[2]	(18.7)
MRNF closure bond deposit[3]	(5.4)
Total net capital cost	**73.8**

11. The total can vary due to rounding.
12. The Corporation expects revenues during the pre-production period.
13. Net amount including the closure bond currently deposited with the MRNF for 5.4 M$.

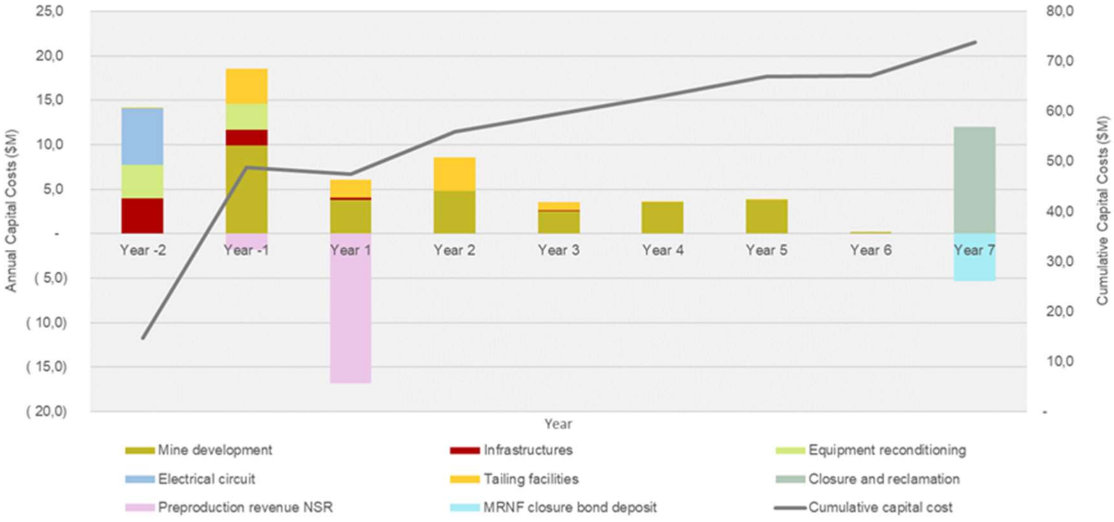


Figure 1.1 – Overall Sleeping Giant capital cost profile

1.14.2 Operating costs

The average operating cost over the six-year mine life is estimated to be $320.67/t mined or $1260/oz (CAD).

The operating unit costs were calculated over the total mineralized material mined from development and from production, excluding the tonnage during preproduction. Table 1.6 provides the breakdown of the projected operating costs for the Project.

Table 1.6 – Sleeping Giant operating cost summary

Area cost description	LOM Operating Cost ($M)	LOM Average Operating Cost per year ($M)	LOM Average Operating Unit Cost ($/t)	LOM Average Operating Unit Cost ($/oz)	OPEX (%)
Definition Drilling	10.8	1.9	15.86	62.3	5%
Stopes Preparation	19.4	3.4	28.45	111.8	9%
Mine Production	67.9	11.8	99.79	392.0	31%
Mine Services	39.7	6.9	58.32	229.1	18%
Maintenance	16.2	2.8	23.81	93.5	7%
Technical Services	19.2	3.3	28.19	110.8	15%
Mill Processing	32.3	5.6	47.17	186.5	1%
Environment	1.2	0.2	1.76	6.9	5%
General and Administration (“G&A”)	11.6	2.0	17.01	66.8	15%
Total	**218.3**	**37.9**	**320.67**	**1259.7**	**100%**

The operating cost estimate was based on Q1 2023 assumptions. The estimate has been deemed to be of an accuracy within alignment with a PEA level of study. The operating cost estimate is based on the mine schedule indicative tonnage per period that was produced by InnovExplo in May 2023. See Chapter 16 for more details related to the mine plan.

1.15 Economics Analysis

The economic assessment of the Project was carried out using a discounted cash flow approach on a pre-tax and after-tax basis, based on Q1 2023 metal price projections in USD and cost estimates of CAPEX and operating expenditures (“OPEX”) in CAD. Inflation or cost escalation factors were not considered. The base-case gold price is 1,800 USD/oz.

There are several known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here. The 2023 PEA is a preliminary assessment and includes the use of Inferred mineral resources that are considered too speculative geologically to be considered mineral reserves due to the lack of economic considerations applied to them.

The base case financial model (after-tax) resulted in an IRR of 33.3% and a NPV of $54.4M using a 5% discount rate. The after-tax payback period after start of operations is 2.2 years.

The base case financial model (pre-tax) resulted in an IRR of 41.4% and a NPV of $77.5M using a 5% discount rate. The after-tax payback period after start of operations is 2.1 years.

The financial results and parameters are presented in Table 1.7.

Table 1.7 – Financial base parameters

Description	Unit	Value
Gold price	USD/oz	1,800
Exchange rate	USD:CAD	0.77
Discount rate	%	5
Mine life (including pre-production, reclamation and closure period)	Years	8
Total mined	t (000)	720
Gold grade	g/t	8.10
Process overall gold recovery	%	96.7
Operating mining cost	$/t mined	320.67
Royalties	% NSR	2
Pre-production capital cost[1]	$M	42.0
Sustaining capital cost[2]	$M	31.8

1. Includes capitalised revenues generated during pre-production period;
2. Include a closure bond of $5.4 M held by the MRNF

1.16 Interpretations and Conclusions

The 2023 PEA was prepared by the InnovExplo consulting team, in collaboration with WSP, Soutex and Englobe, to demonstrate the economic viability of re-opening and developing the Sleeping Giant mine. The Report provides a summary of the results and findings from each major area of investigation. It was carried out according to standard industry practices, equipment, and processes. According to the available information, the Report Authors are unaware of any unusual or significant risks or uncertainties that could materially affect the Project's reliability or confidence.

Based on the base-case assumptions, the 2023 PEA found the proposed Project to be technically and financially feasible. Several additional studies, such as field work, drilling campaign, metallurgical testing, and trade-off analyses, are needed to support more advanced mining studies. The Report Authors consider the 2023 PEA results sufficiently reliable and recommend initiation of a feasibility study as the next step in the Project's development.

1.16.1 Risks and Opportunities

An analysis of the results has identified a series of risks and opportunities associated with each of the technical aspects considered for the development of the Project.

Potential Risks

The most significant potential risks associated with the Project are:

- Locally inaccurate historical assay results, and local bias in grade estimates.
- Locally poor geological continuity of veins (i.e., local dislocations, pinch, and swell); Poor geological continuity could negatively affect the accurate localization of mineralized blocks and/or grade estimation.
- Local uncertainties about historical mine openings and local bias in estimates.
- Old underground infrastructure: rehabilitation work is needed to restart operations. Inadequacies have been identified in the existing infrastructure.
- Lower gold recovery than predicted that will negatively impact the Project's profitability
- Risks associated with water management at the TSF
- Social acceptability related to new or modified infrastructures (i.e. camp site, wastewater treatment facility) is an inherent risk for all mining projects. It could affect the Project's permitting and development schedule.
- Inability to attract experienced professionals; The ability to attract and retain competent, experienced professionals is a key factor to success

Many of the previous noted risks are common to most mining projects, many of which may be mitigated, at least to some degree, with adequate engineering, planning and pro-active management.

Key Opportunities

The Report identifies several opportunities that have the potential to enhance the economics, timing, and permitting aspects of the Project. The key opportunities currently identified are as follows:

- Further 3D modelling and interpretation using current drill hole data;
- Drill areas with strong possibilities of expanding known mineralized zones;
- Understand the structural pattern in certain areas;
- Reduce the thickness of the surface pillar;
- Include channel samples in the interpolation procedure to potentially improve the block classification in certain areas;
- Refine the 3D solids used for mining depletion;
- Optimize logging procedures;
- Relogging and resampling of drill holes;
- Process ore from other properties to increase feed tonnage to reach plant capacity of 750 tpd;
- Recovery of economic material at the vicinity of legacy mined areas;
- Optimize water and tailings management.

1.17 Recommendations

To summarize, the Report Authors advise moving forward with the Project to the feasibility study phase. They also suggest ongoing environmental and permitting efforts to align with Sleeping Giant development plans and Project schedule. Based on the recommendations of these QP, a work program has been devised that includes further exploration drilling and the feasibility study. The estimated cost of the work program is approximately $4.6 million, with a 15 % contingency (Table 1.8).

For a detailed breakdown of the estimated budget, refer to Table 1.8. Furthermore, Chapter 26 provides additional recommendations.

Table 1.8 – Estimated Costs for the Recommended Work Program

WORK PROGRAM	BUDGET COST
Exploration and definition drilling (approx. 23,000 m at $150/m)	$3,450,000
Implement a QA/QC data validation system	$30,000
Channel sampling 3D localization	$20,000
Electrical distribution modifications (Addition of ground fault protection)	$447,000
H-Q power line modification for camp module	$160,000
Optimize mine plan	$50,000
Continue / start permitting initiatives related to the Cell 2A of the TSF to advance toward production	$10,000
TSF: Trade-off study between lift 5 in Cell 2A and a new cell (Cell 4)	$30,000
TSF: Start feasibility study related to the Cell 2A	$50,000
TSF: Engineering/operational procedures	$50,000
Wetland delineation related to the camp infrastructure	$15,000
Groundwater wells: hydrogeological study, tendering and permitting	$150,000
Wastewater treatment system: design, permitting, and specifications	$45,000
Hydrogeological study in connection with the depollution attestation	$140,000
TOTAL	$4,647,000

2. INTRODUCTION

Mines Abcourt Inc. (“Abcourt” or the “Issuer”) retained InnovExplo Inc. (“InnovExplo”) to create a Pre-economic assessment (the “2023 PEA”), of the Sleeping Giant Project latest resource estimate (the “2022 MRE”), and a supporting Report (the “Report”).

Pascal Hamelin, President and CEO of Abcourt, assigned the mandate.

Abcourt acquired the Property in 2016. The Project was an active underground mine that is now under care and maintenance. Abcourt used the onsite processing plant to mill material from its Elder Project. The Project has all the required infrastructure for a potential restart.

This Report has been prepared in accordance with Canadian Securities Administrators’ National Instrument 43-101 Respecting Standards of Disclosure for Mineral Projects (“NI 43 101”) and its related Form 43 101F1.

The 2022 MRE has an effective date of December 12, 2022. It represents an update of the feasibility study published by Bonneville in 2019 (the “2019 FS”).

The 2023 PEA has an effective date of June 07, 2023. It represents an assessment of the 2022 MRE by Vadnais-Leblanc and Lecomte.

InnovExplo is an independent geology and mining engineering consulting firm based in Val-d’Or (Quebec, Canada), with other provincial offices in Quebec City and Longueuil. Outside of these offices, InnovExplo also employs professional consultants in Montreal, Trois-Rivières (Quebec, Canada) and Sudbury (Ontario, Canada).

2.1 Issuer

Abcourt is a Canadian mining company trading publicly on the TSX Venture Exchange under the symbol ABI (TSXV:ABI), the Berlin Stock Exchange under the symbol AML BE, and the Frankfurt Exchange under the symbol AML-FF. Its head office is located at 475, avenue de l’Église, Rouyn-Noranda, Québec, J0Z 1Y1, Canada.

Abcourt was constituted by letters patent of amalgamation in January 1971 and continued its existence under Part 1A of the Quebec Companies Act in March 1981. On February 14, 2011, Abcourt was automatically continued under the Business Corporations Act (Quebec) after this law entered into force. Abcourt is engaged in acquiring, evaluating, evaluating and exploiting mining properties in Canada, principally for gold.

The Property consists of one contiguous block of 132 claims converted into map-designated cells (or “CDC”) and four (4) mining leases, collectively covering an aggregate area of 6480.4 ha, in the townships of Maizerets, Glandelet, Soissons and Chaste on NTS map sheets 32E01 and 32F04. All claims and leases are registered 100% to Mines Abcourt inc. A detailed claim and lease list is provided in Appendix I.

Abcourt is the sole owner of all leases and mining titles on the Property.

2.2 Terms of References

The Report presents and supports an updated mineral resource estimate for the Project. Most of the supporting information was gleaned from underground drilling and development of gold-bearing rock at the mine, supplemented by data from previous companies that conducted drilling in the area constituting the current Property. Historical details, geological information (local and regional) and general information relevant to the mine are also described.

The 2022 MRE and the 2023 PEA have been prepared by InnovExplo's independent qualified persons (QP) for the Issuer. The 2022 MRE and 2023 PEA adhere to the current Canadian Reporting Standards for Mineral Resources and Mineral Reserves, which are the Canadian Institute of Mining Metallurgy and Petroleum ("CIM") Definition Standards for Mineral Resources and Mineral Reserves of May 2014 ("CIM Definition Standards"). The 2022 MRE and 2023 PEA also follows the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines of November 2019 (the "CIM MRMR Best Practice Guidelines").

2.3 Principal Source of Information

As part of the mandate, InnovExplo has reviewed the following with respect to the Project: the mining titles and their status on the GESTIM website (the Government of Québec's online claim Abcourtent system); agreements and technical data supplied by the Issuer (or its agents); and the Issuer's filings on SEDAR (press releases and MD&A reports).

InnovExplo has no known reason to believe that any information used to prepare this Report is invalid or contains misrepresentations. The authors have sourced the information for the Report from the reports listed in Item 27.

InnovExplo reviewed and appraised the information used to prepare the Report, including the conclusions and recommendations. InnovExplo believes this information is valid and appropriate, considering the status of the Project and the purpose for which the Report is prepared.

This 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 Report.

2.4 Report Responsibility and Qualified Persons

InnovExplo's independent QP, as defined in NI 43 101, prepared the Report and the 2023 PEA. The QP for the 2023 PEA are Olivier Vadnais-Leblanc, P.Geo. and Éric Lecomte, P.Eng. Table 1.1 lists the QP for the Report and the sections for which each QP is responsible.

The individuals listed in Table 1.1, by virtue of their education, experience, and professional association, are considered Qualified Persons (QP) as defined in the NI 43-101 and are members in good standing of appropriate professional institutions.

The QP have contributed to the writing of the Report and have provided QP certificates, included at the beginning of the Report. The information contained in the certificates outlines the sections in the Report for which each QP is responsible. Each QP has also contributed figures, tables, and parts of Chapter 1 (Summary), Chapter 2 (Introduction), Chapter 25 (Interpretation and Conclusions), Chapter 26 (Recommendations), and Chapter 27 (References).

Table 2.1 – Report contributors

Qualified Person	Company	Areas of report responsibility
Eric Lecomte	InnovExplo Inc (“InnovExplo”)	Items 14.1.11, 14.2.2, 16, 19, 22 and 24 & Contributor for items 1, 2, 3, 21, 25, 26 and 27.
Olivier Vadnais-Leblanc	InnovExplo Inc (“InnovExplo”)	Items 4 to 12, 14.1.1 to 14.1.10, 14.1.12 to 14.1.13, 14.2.1, 14.3 and 23 & Contributor for items 1, 2, 3, 25, 26 and 27.
Luc Boutin	WSP Inc (“WSP”)	Items 18.1 to 18.14.1 and 18.17 & Contributor for items 1, 2, 21, 25, 26 and 27.
Guy Comeau	Soutex Inc (“Soutex”)	Items 13 and 17 & Contributor for items 1, 2, 3, 21, 25 and 26.
Jacques Blanchet	Englobe Corp (“Englobe”)	Items 18.14.2, 18.14.3 and 18.18 to 18.20 & Contributor for items 1, 2, 21.1 and 25 to 27.
Marc L’Écuyer	Englobe Corp (“Englobe”)	Items 18.15, 18.16 and 20 & Contributor for items 1, 2, 21 and 25 to 27.

The QP do not have, nor 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 consulting firm.

2.5 Site Visit

The following bulleted list describes which QP visited the Property, the date of the visit, and the general objective of the visit:

- Eric Lecomte of InnovExplo has visited the Property on February 7th, 2023;
- Olivier Vadnais-Leblanc of InnovExplo visited the Property on November 7th and 8th, 2022, for the purpose of this mandate. The site visit included an underground tour and a visual assessment of surface infrastructures, access, mill, laboratory, office and core shack facilities;
- Luc Boutin of WSP visited the Property on February 7th and on April 27th, 2023 for the purpose of this mandate. The site visits included a walk through the administration building, the warehouse, the maintenance shop, the dry, the ore treatment plant and all electrical rooms and the hoist rooms;
- Guy Comeau of Soutex visited the Property on February 7th, 2023 for the purpose of this mandate. The site visit included a visual assessment of the mill, laboratory and the office;
- Jacques Blanchet of Englobe has not visited the Property for the purpose of this Technical Report.
- Marc L’Écuyer of Englobe has not visited the Property for the purpose of this Technical Report.

2.6 Effective Date

The effective date of the 2022 MRE and the Report is December 12, 2022.

The effective date of the Report is June 07, 2023.

The signature date is July 19, 2023.

The close-out date of the drill hole database is June 10, 2022, including all available drilling data. No drilling was in progress while the mineral resource estimate was being prepared.

The Report was prepared as a National Instrument 43-101 Standards of Disclosure for Mineral Projects Report for Abcourt by QP, collectively the "Report Authors". The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Authors' services, based on:

- Information available at the time of preparation;
- Data supplied by outside sources;
- The assumptions, conditions, and qualifications set forth in the Report.

The Report is intended for use by Abcourt, subject to the terms and conditions of its respective contracts with the Report Authors, and relevant securities legislation.

The contract allows Abcourt to file the Report with Canadian Securities Regulatory Authorities pursuant to National Instrument 43-101. Except for the purposes legislated under provincial securities law, any other uses of the Report by any third party are at that party's sole risk. The responsibility for this disclosure remains with Abcourt. The user of this document should ensure that this is the most recent Report for the Property, as it is not valid if a new Report has been issued.

2.7 Sources of Information

2.7.1 General

The Report is based in part on internal company reports, maps, published government reports, company letters and memoranda, and public information, as listed in Chapter 27 (References) of the Report.

Sections from reports authored by other consultants may have been directly quoted or summarized in the Report and are so indicated, where appropriate.

The 2023 PEA has been completed using available information contained in, but not limited to, the following reports, documents, and discussions:

- Technical discussion with Abcourt personnel;
- QP personal inspection of the Property;
- Reports of mineralogical and metallurgical characteristics;
- The Project resource block model and estimate provided by InnovExplo, which are effective as of June 10, 2022;
- The Project revised Environmental Assessment;
- Various reports covering site hydrology, hydrogeology, geotechnical, and geochemistry;
- Additional information from public domain sources.

The Report Authors reviewed and appraised all the information used to prepare the Report and believe that such information is valid and appropriate considering the status of the Project and the purpose for which the Report is prepared. The Report Authors have thoroughly researched and documented their conclusions and recommendations made in the Report.

2.7.2 InnovExplo Inc.

The following entities or individuals provided specialist input to Eric Lecomte, QP:

- Sebastien Tanguay, senior mining engineer, is responsible for the geotechical and ground control considerations.
- Jean-Olivier Brassard, mining engineer, is responsible for the Deswik optimizing process, design and scheduling;

2.7.3 WSP Inc.

The following entities or individuals provided specialist input to Luc Boutin, QP:

- Sedik Bendaoud is a mechanical engineer;
- Olivier Perrault is a structural engineer;
- Pierre-Marc Aubin is a mechanical technician.

2.7.4 Soutex Inc.

The following entities or individuals provided specialist input to Guy Comeau QP:

- Simon Fortier is a process engineer.

2.7.5 Englobe Corp.

The following entities or individuals provided specialist input to Jacques Blanchet, QP:

- Philippe Rio Roberge is a civil engineer;
- Lydia Charbonneau is a civil engineer.

The following entities or individuals provided specialist input to Marc L’Écuyer, QP:

- Sylvain Arseneault is a biologist;
- Martin Pérusse is a biologist.

2.8 Currency, Units of Measure, and Acronyms

The abbreviations, acronyms and units used in the Report are provided in Table 2.2 and Table 2.3. 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 certain elemental grades, and gram per metric ton (g/t) for precious metal grades. Wherever applicable, imperial units have been converted to the International System of Units (SI units) for consistency (Table 2.4).

Table 2.2 – List of Acronyms

Acronyms	Term
43-101	National Instrument 43-101 (Regulation 43-101 in Québec)
Ai	Abrasion index
AMIS	Abandoned Mines Information System
ASTM	American Society for Testing and Materials
APR	Annual percentage rate
BAPE	Bureau d'audience publique du Québec
BWi	Bond work index
CofA	Certificate of authorization
CA	Core angle
CAD:USD	Canadian-American exchange rate
CNSC	Canadian Nuclear Safety Commission
CAPEX	Capital expenditure
CDPNQ	Centre de données sur le patrimoine naturel du Québec
CEAA 2012	Canadian Environmental Assessment Act (2012)
CEAAg	Canadian Environmental Assessment Agency
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIM Definition Standards	CIM Definition Standards for Mineral Resources and Mineral Reserves
CL	Core length
CMS	Cavity monitoring system
CoG	cut-off grade
CRM	Certified reference material
CSA	Canadian Securities Administrators
CSS	Contact support services
CV	Coefficient of variation
CWi	Crusher work index
DEM	Digital elevation model
DDH	Diamond drill hole
Directive 019	Directive 019 sur l'industrie minière
EA	Environmental assessment
EBITDA	Earnings before interest, taxes, depreciation and amortization
ECA	Environmental Compliance Approval
ECCC	Environment and Climate Change Canada
EDO	Effluent discharge objectives
EEM	Environmental Effects Monitoring
EIA	Environmental impact assessment
EIS	Environmental impact study

Acronyms	Term
EPCM	Engineering, procurement, construction, Abcourtent
EQA	Environment Quality Act
ESA	Environmental site assessment
ESIA	Environmental and social impact assessment
F_{100}	100% passing - Feed
F_{80}	80% passing - Feed
FIFO	Fly in fly out
FOB	Freight on board
FS	Feasibility study
FWR	Fresh water reservoir
G&A	General and administration
GESTIM	Gestion des titres miniers (the MERN’s online claim Abcourtent system)
GHG	Greenhouse gas
GPR	Ground penetrating radar
ID2	Inverse distance squared
ID3	Inverse distance cubed
ID6	Inverse distance power six
IDW	Inverse distance weighting
IEC	International Electrotechnical Commission
IRR	Internal rate of return
ISA	Inter-ramp slope angle
ISO	International Organization for Standardization
ISRM	International Society for Rock Mechanics
IT	Information technology
JBNQA	James Bay and Northern Québec Agreement
JV	Joint venture
JVA	Joint venture agreement
LLC	Limited liability company
LOM	Life of mine
LOMP	Life of mine plan
LUP	Land Use Permit
MACRS	Modified accelerated cost recovery system
MCC	Ministère de la Culture et des Communications du Québec (Québec’s Ministry of Culture and Communications)
MCCCF	Former name of the MCC
MELCCFP	Ministère de l'Environnement, de la Lutte contre les changements climatiques, de la Faune et des Parcs (Québec’s Ministry of Environment, the Fight Against Climate Change, the Wildlife and Parks)

Acronyms	Term
MDI	Mineral Deposit Inventory
MRNF	Ministère des Ressources naturelles et des Forêts du Québec (Québec's Natural Resources and Forests)
mesh	US mesh
MIK	Multiple indicator kriging
MLO	Mining Licence of Occupation
MMER	Metal mining effluent regulations
MNDM	Ontario Ministry of Northern Development and Mines
MNR	Ontario Ministry of Natural Resources
MRC	Municipalité régionale de comté (Regional county municipality in English)
MRE	Mineral resource estimate
MRMR	Mineral resources and mineral reserves
MRN	Former name of MERN
MSHA	Mine Safety & Health Administration
MSO	Mineable Shape Optimizer
MTSMTE	Ministère des Transports, de la Mobilité durable et de l'Électrification des transports du Québec (Québec's Ministry of Transport, Sustainable Mobility and Transport Electrification)
MWMP	Meteoric water mobility potential
n/a	Not applicable
N/A	Not available
NAD	North American Datum
NAD 27	North American Datum of 1927
NAD 83	North American Datum of 1983
nd	Not determined
NI 43-101	National Instrument 43-101 (Regulation 43-101 in Québec)
NN	Nearest neighbour
NPI	Net profits interest
NPV	Net present value
NRC	Natural Resources Canada
NSR	Net smelter return
NTS	National Topographic System
OER	Objectifs environnementaux de rejet (Québec)
OK	Ordinary kriging
OPEX	Operational expenditure
P_{80}	80% passing - Product
P_{100}	100% passing - Product
PAG	Potentially acid generating

Acronyms	Term
PFS	Prefeasibility study
PM	Particulate matter
PMF	Probable maximum flood
PMP	Probable maximum precipitation
POF	Probability of failure
Q	Value expressing quality of rock mass (Q-system for rock mass classification)
QA	Quality assurance
QA/QC	Quality assurance/quality control
QBBA	Québec Breeding Bird Atlas
QC	Quality control
QP	Qualified person (as defined in National Instrument 43-101)
R&D	Research and development
RBQ	Régie du Bâtiment du Québec
RC	Reverse circulation (drilling)
Regulation 43-101	National Instrument 43-101 (name in Québec)
RMR	Rock mass rating
ROM	Run of mine
RQD	Rock quality designation
RQI	Rock quality index
RWi	Rod work index
SABC	Comminution circuit consisting of a SAG mill, ball mill and pebble crusher
SAG	Semi-autogenous-grinding
SARA	Species at Risk Public Registry
SCC	Standards Council of Canada
SD	Standard deviation
SF	Safety factor
SG	Specific gravity
SIGÉOM	Système d'information géominière (the MERN's online spatial reference geomining information system)
SMC	SAG mill comminution
SMU	Selective mining unit
SPLP	Synthetic Precipitation Leaching Procedure
TCLP	Toxicity characteristic leaching procedure
TDS	Total dissolved solids
TMF	Tailings Abcourtent facility
TSP	Total suspended particulate matter
UCoG	Underground cut-off grade

Acronyms	Term
UCS	Uniaxial compressive strength
UG	Underground
UTM	Universal Transverse Mercator coordinate system
VOD	Ventilation on demand
WBS	Work breakdown structure
WSR	Water storage reservoir

Table 2.3 – List of units

Symbol	Unit
%	Percent
% solids	Percent solids by weight
$, C$	Canadian dollar
$/t	Dollars per metric ton
°	Angular degree
°C	Degree Celsius
µm	Micron (micrometre)
µS/cm	Micro-siemens per centimetre
A	Ampere
avdp	Avoirdupois
Btu	British thermal unit
cfm	Cubic feet per minute
cfs	Cubic feet per second
cm	Centimetre
cm^2	Square centimetre
cm^2/d	Square centimetre per day
cm^3	Cubic centimetre
cP	Centipoise (viscosity)
d	Day (24 hours)
dm	Decametre
ft	Foot (12 inches)
g	Gram
G	Billion
Ga	Billion years
gal/min	Gallon per minut
g-Cal	Gram-calories
g/cm^3	Gram per cubic centimetre
g/L	Gram per litre

Symbol	Unit
g/t	Gram per metric ton (tonne)
GW	Gigawatt
h	Hour (60 minutes)
ha	Hectare
hp	Horsepower
Hz	Hertz
in	Inch
k	Thousand (000)
ka	Thousand years
kbar	Kilobar
kg	Kilogram
kg/h	Kilogram per hour
kg/t	Kilogram per metric ton
kj	Kilojoule
km	Kilometre
km^2	Square kilometre
km/h	Kilometres per hour
koz	Thousand ounces
kPa	Kilopascal
kW	Kilowatt
kWh	Kilowatt-hour
kWh/t	Kilowatt-hour per metric ton
kVA	Kilo-volt-ampere
L	Litre
lb	Pound
lb/gal	Pounds per gallon
lb/st	Pounds per short ton
L/h	Litre per hour
L/min	Litre per minute
lbs NiEq	Nickel equivalent pounds
M	Million
m	Metre
m^2	Square metre
m^3	Cubic metre
m/d	Metre per day
m^3/h	Cubic metres per hour
m^3/min	Cubic metres per minute

Symbol	Unit
m/s	Metre per second
m^3/s	Cubic metres per second
Ma	Million years (annum)
masl	Metres above mean sea level
Mbgs	Metres below ground surface
Mbps	Megabits per second
MBtu	Million British thermal units
mi	Mile
min	Minute (60 seconds)
Mlbs	Million pounds
ML/d	Million litres per day
mm	Millimetre
mm^2	Square millimetres
mm Hg	Millimetres of mercury
mm WC	Millimetres water column
Moz	Million (troy) ounces
mph	Mile per hour
Mt	Million metric tons
MW	Megawatt
ng	Nanogram
NiEq	Nickel equivalent
oz	Troy ounce
oz/t	Ounce (troy) per short ton (2,000 lbs)
ppb	Parts per billion
ppm	Parts per million
psf	Pounds per square foot
psi	Pounds per square inch
rpm	Revolutions per minute
s	Second
s^2	Second squared
scfm	Standard cubic feet per minute
st/d	Short tons per day
st/h	Short tons per hour
t	Metric tonne (1,000 kg)
ton	Short ton (2,000 lbs)
tpy	Metric tonnes per year
tpd	Metric tonnes per day

Symbol	Unit
tph	Metric tonnes per hour
US$	American dollar
usgpm	US gallons per minute
V	Volt
vol%	Percent by volume
wt%	Weight percent
y	Year (365 days)
yd^3	Cubic yard

Table 2.4 – 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 Qualified Persons ("QP") have relied on reports, sources of information and opinions provided by the Issuer regarding the mineral rights, surface rights, ownership agreements, royalties, and tax status of the Sleeping Giant Project ("the Project"). The QP are not qualified to express any legal opinion with respect to property titles, current ownership or possible litigation.

At the date of this Technical Report titled “Preliminary Economic Assessment for the Sleeping Giant Project” ("the Report"), Abcourt indicates that there are no known disputes likely to affect the Project.

A preliminary version of the Report has been reviewed by Abcourt for factual errors. Any changes made in response to the review have not altered the conclusions reached. Accordingly, the statements and opinions expressed herein are given in good faith and with the belief that such statements and opinions are neither false nor misleading as at the issue date of the Report.

3.1 Taxation and Royalties

The Project had been evaluated on an after-tax basis to provide an approximate value of the potential economics. The taxation for the Project was compiled with the assistance of finance experts Raymond Chabot Grant Thornton.

Calculations are based on Canada tax regime as at the date of the Report. A federal tax of 15 % and a Quebec tax rate of 11.5 % has been used for calculation. A royalty of 2 % has been assumed for the Project.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Project is located 80 km north of the town of Amos in northwestern Quebec (Figure 4.1). It is accessed via Quebec provincial highway Route 109, which runs through the Property and connects Amos to Matagami. A 1-km gravel road leads from the paved highway to the mine.

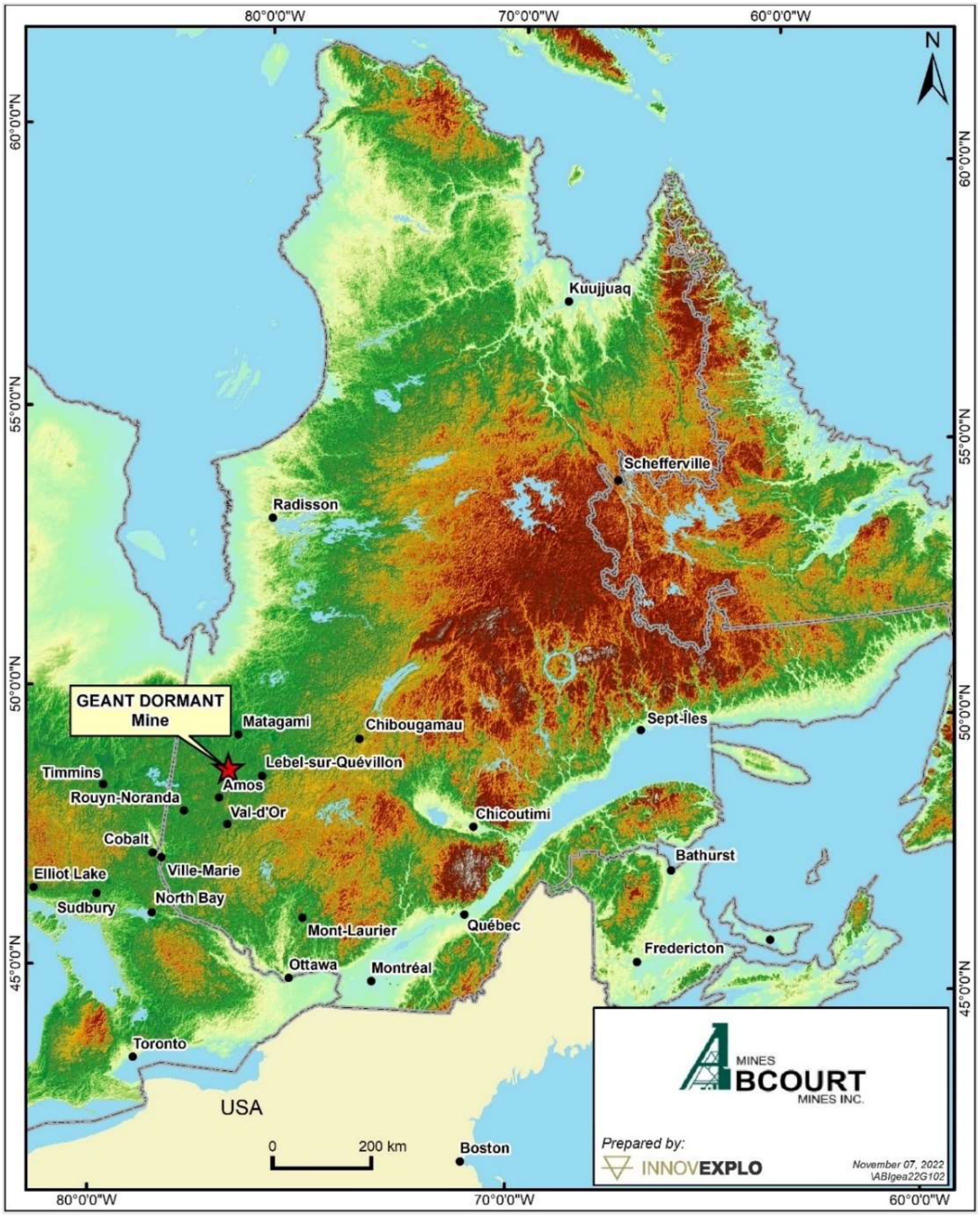


Figure 4.1 – Provincial map showing the location of the Sleeping Giant Property and mine (a.k.a., Géant Dormant)

4.2 Mining Rights in Québec

The following information on the mining rights in Quebec was mostly taken from Guzun (2012), Gagné and Masson (2013), and from the *Act to amend the Mining Act* (Bill 70) assented to on December 10, 2013 (National Assembly, 2013).

In Quebec, mining is principally regulated by the provincial government. The MERN is the provincial agency entrusted with managing mineral substances in Quebec. The ownership and granting of mining titles for mineral substances are primarily governed by the *Mining Act* and related regulations. In Quebec, land surface rights are distinct property from mining rights. Rights in or over mineral substances in Quebec 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 the *Mining Act* governs certain aspects of the exploration for mining of such mineral substances.

4.2.1 Claims

A claim is the only exploration title for mineral substances (other than surface mineral substances, petroleum, natural gas and brine) currently issued in Quebec. 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. 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 selects available pre-mapped claims online. Claims can be obtained by staking in a few government-defined areas.

A claim has a term of two years, renewable for additional two-year periods, subject to the performance of minimum exploration work on the claim and compliance with other requirements set forth by the *Mining 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, the claim holder can 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, the *Mining Act* requires a claim holder to submit to the Minister, on each anniversary date of the claim registration, 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 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 (6) subsequent renewal periods (12 years in total). Holders of a mining lease or a mining concession cannot apply any work carried out in respect of a mining lease or a mining concession to claim renewals.

The *Mining Act* (as amended) requires a claim holder to notify the following within 60 days after registering a claim: the owner, the lessee, the holder of an exclusive lease to mine surface mineral substances, and the local municipality where the claim is located. A claim holder must also notify the municipality and landowner of work to be carried out on its claim at least 30 days before performing the work.

4.2.2 Mining leases

Mining leases and 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). A mining lease is granted to the holder of one or several claims upon proof of the existence of indicators of a workable deposit in 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.

The *Mining Act* (as amended) states that an application for a mining lease must be accompanied by a project feasibility study and by a scoping and market study as regards to processing in Quebec. Holders of mining leases must produce such a scoping and market study every 20 years thenceforth. The *Mining Act* adds, as an additional condition for granting a mining lease, the issuance of the certificate of authorization under the *Environment Quality Act*. The Minister may nevertheless grant a mining lease if the time required to obtain the certificate of authorization is unreasonable. The Minister must approve a rehabilitation and restoration plan 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. The *Mining Act* stipulates that the financial guarantee to be provided by a mining lease holder shall be for an amount corresponding to the anticipated total cost of completing the work required under the rehabilitation and restoration plan.

4.2.3 Other information

Claims, mining leases and 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 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 the *Mining Act*, 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 the same period, as well as any other information as determined by regulation.

4.3 Mining Titles and Claim Status

Abcourt acquired the Property on June 20, 2016, from Deloitte (sequestered mining assets from Aurbec bankruptcy; see Item 6.6 for details) for a cash sum of $2,548,727 and a royalty. Abcourt is the sole owner of all leases and mining titles on the Property.

InnovExplo retrieved the status of the Property's claims and leases from GESTIM, the Government of Quebec's online claim management system (http://gestim.mines.gouv.qc.ca). According to GESTIM, all mining titles were in good standing on October 17, 2022, the date of the data retrieval.

The Property consists of one contiguous block of 132 claims converted to map-designated cells (or "CDC") and four (4) mining leases, collectively covering an aggregate area of 6480.4 ha (Figure 4.2) in the townships of Maizerets, Glandelet, Soissons and Chaste on NTS map sheets 32E01 and 32F04. The claims and mining lease are registered 100% to Mines Abcourt inc. A detailed list is provided in Appendix I.

4.4 Royalties

The Property is subject to a royalty of $5.00 per tonne (t) to Cyrus Capital Partners L.P. on the first 350,000 t extracted from the Sleeping Giant mine. Cyrus Capital Partners manages holdings through FBC Holdings., a Luxembourg-based limited liability company. All other prior royalties were purged under the 2016 judgment related to Aurbec's bankruptcy.

In a press release dated September 22, 2022, Abcourt announced it had granted Maverix Metals Inc. a 2 % NSR royalty ("the Royalty") on all metallic and non-metallic minerals mined or otherwise recovered on each of the Sleeping Giant and Dormex properties, in accordance with the terms of an NSR royalty agreement. Maverix Metals made a US$2,000,000 cash payment to Abcourt as consideration for the Royalty.

Abcourt may, at any time prior to the third (3rd) anniversary of the transaction, elect to reduce the Royalty rate by 0.5% upon payment of an amount of US$2,000,000, and may, at any time prior to the sixth (6th) anniversary of the transaction, elect to reduce the Royalty rate by 0.5 % upon payment of an amount of US$4,000,000 to the holder of the Royalty, thereby reducing the Royalty rate to 1.0%, to the extent Abcourt also exercises the first option.

The QP are not aware of other royalties granted on leases and other mining titles.

4.5 Environment and Permits

Abcourt has all the necessary authorizations and permits to conduct exploration works, underground mine care and maintenance, mill cleaning and refurbishment, and other exploration and mining activities on the Property's mining leases.

Abcourt holds the surface rights of the mining leases.

InnovExplo is not aware of any environmental or social issues concerning the Property. All exploration activities conducted on the Property comply with relevant environmental permitting requirements. To InnovExplo's knowledge, Abcourt has obtained the appropriate permits relating to the surface rights for its milling and waste disposal facilities and for underground mining activities.

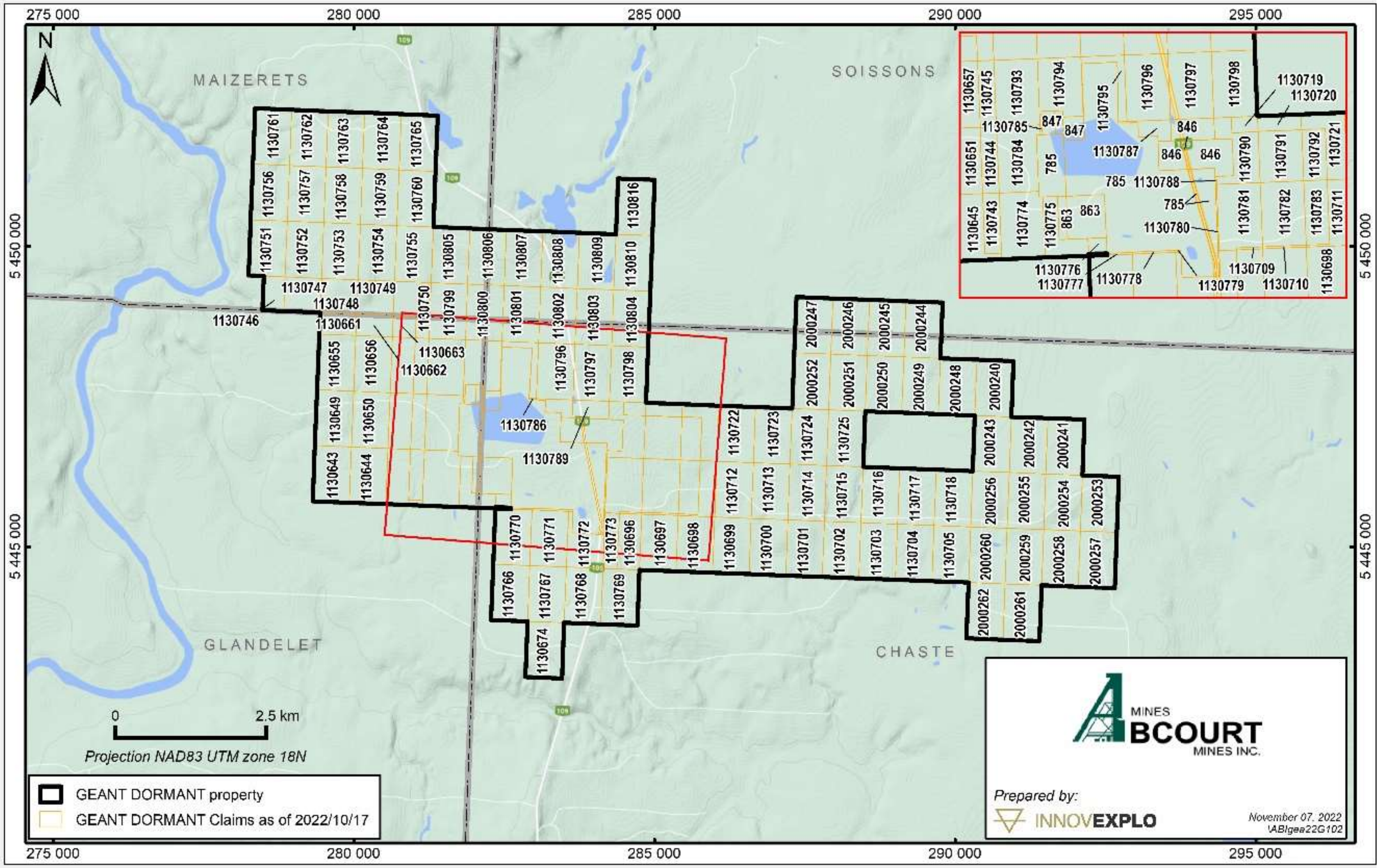


Figure 4.2 – Mining Leases and Claims

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

5.1 Accessibility

The Property is easily accessible from Amos by driving 80 km north on Quebec provincial highway Route 109, which connects Amos (pop. 12,800) to Matagami (pop. 1,450). A 1-km gravel road leads from the paved highway to the mine, and a network of forestry roads leads to other parts of the Property (Figure 5.1). The landscape is relatively flat and lightly wooded. It is bounded to the west and south by the Harricana and Coigny rivers, respectively. The thickness of the overburden varies between 15 and 60 m, for an average of 30 m.

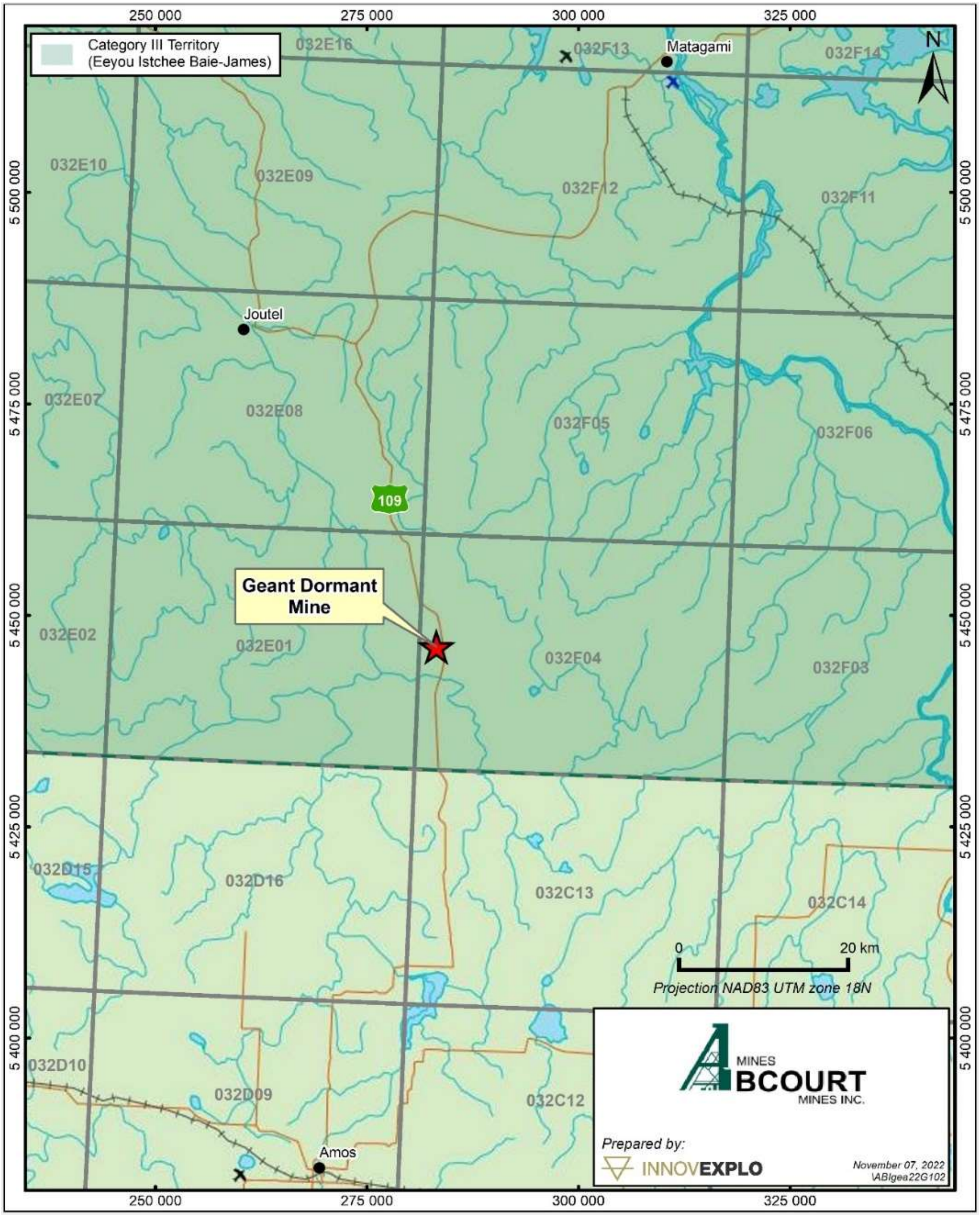
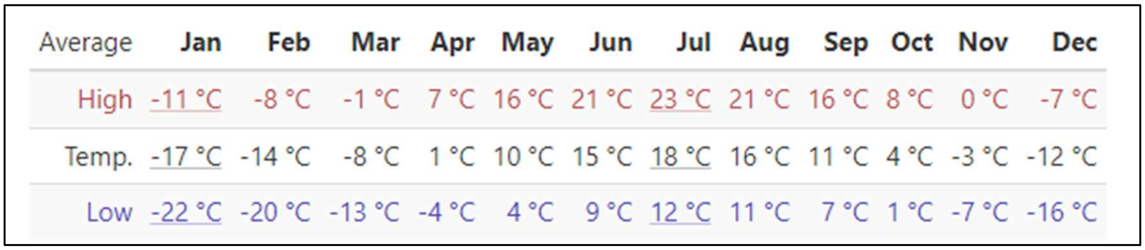


Figure 5.1 – Base map showing local water sources and access routes to the Property

5.2 Climate

The region experiences a continental climate marked by cold and dry winters and hot, humid summers. In Amos, the summers are long, comfortable, and partly cloudy, and the winters are frigid, snowy, and overcast. Over the course of a year, the temperature typically varies from -22°C to 23 °C and is rarely below -34 °C or above 29 °C. The average temperature in July is 16.8 °C, and in January, it is around -17.5 °C (Table 5.1). On average, there are 209 frost days during the year. The historical records of annual precipitation show an average rainfall of 929 mm in the form of rain and snow. Rainfall rates are stable at ±105 mm per month in summer, whereas snowfall rates reach ±50 cm per month in winter. Sunshine hours peak in July (249 hrs) and are at their lowest in November (53 hrs). Exploration work can be done year-round on the Property, except during the spring thaw period.

Table 5.1 – Average Monthly Temperatures in Amos



Average	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
High	-11 °C	-8 °C	-1 °C	7 °C	16 °C	21 °C	23 °C	21 °C	16 °C	8 °C	0 °C	-7 °C
Temp.	-17 °C	-14 °C	-8 °C	1 °C	10 °C	15 °C	18 °C	16 °C	11 °C	4 °C	-3 °C	-12 °C
Low	-22 °C	-20 °C	-13 °C	-4 °C	4 °C	9 °C	12 °C	11 °C	7 °C	1 °C	-7 °C	-16 °C

(https://weatherspark.com/y/20437/Average-Weather-in-Amos-Canada-Year-Round)

5.3 Local Resources

Amos is a medium-sized city with a population of about 12,800 and is the closest major service community. Suppliers, contractors, consulting firms, and skilled and general labour are readily available. The Amos Hospital is recognized for excellence in care offered to the entire population of the Abitibi region. A skilled workforce is available at Amos and in nearby mining towns such as Matagami (pop. 1,450), Lebel-sur-Quévillon (pop. 2,200), Val-d'Or (pop. 32,500), Rouyn-Noranda (pop. 42,200), La Sarre (pop. 7,300) and Chibougamau (pop. 7,500). Forestry and mining industries are important hubs of economic activity in the region. One high-voltage electrical line passes through the Property providing the energy needed for the operation of the processing plant and other mine infrastructures. A railway crosses Abitibi and Northern Ontario via Amos. Even though the Matagami and Amos airports can receive Dash-8-type aircraft, the nearest commercial airport offering regular flights to Montreal is Val-d'Or.

5.4 Infrastructure

The project infrastructure includes underground mine workings, a headframe, milling and processing facilities, a laboratory, workshops and a warehouse, offices, core shack facilities, a tailings facility and basins, stockpile areas, a power line, and a network of surface access roads.

In August 2022, Abcourt commissioned a firm specializing in cleaning gold mills. The firm began cleaning the Sleeping Giant processing plant in September 2022 and was still continuing with the gold recovery process from the cleaning activities as at the effective date of the Report. Abcourt planned to continue the cleaning program until February 2023, when the plant was placed into care and maintenance mode until mining operations can feed the plant at a constant rate.

The Sleeping Giant Mill has a capacity of 750 tpd. Mineralized material can be transported directly to the ore pass feeding the crusher or stored on the stockpile, which has a capacity of 10,000 t. The plant is used for grinding, leaching and desorption, and gold electrolysis and refining. The mill's waste is sent as pulp to the tailings management facility (TMF) through an 8-inch-diameter pipe.

The TMF consists of three ponds. The first is used to decant the pulp, and the second recovers water for reuse as process water in the plant. The surplus is sent to the polishing pond for processing (once a year). Water from this polishing pond is sent to two other ponds, where it is analyzed before being discharged into the environment after ensuring it complies with the rules and by-laws of the MELCCFP (Quebec's Ministry of the Environment).

Hydro-Québec supplies the mine with electricity. The capacity is greater than 5,000 KVA, which is adequate for surface and underground operations.

The ventilation shaft and surface shaft shelters are operational. The service building, mechanical workshop, electrical workshop, equipment warehouse, hoist room, compressor room, generator, water pump for fire, drinking water pumping system, and dry room are all functional as at the effective date of the Report.

The headframe and service shaft provides access to the underground infrastructure. The service shaft accesses 22 levels. The first level is 50 m from the surface, and the lowermost is at a depth of 1,175 m. The ventilation shaft is used to bring fresh air into the mine and can also be used as an emergency exit for employees.

The underground mine pumping system is functional. Underground water is pumped into basins designed for that purpose. The waste pile receives low-grade development material. Non-mineralized rocks on the waste rock pile are mostly used as construction material for dykes in the TMF, as the rock is not acid-generating.

5.5 Physiography

The bedrock underlying the Property is mainly covered with a thick layer of glacial sediment dating from the Pleistocene, although there are a few surface outcrops. Large swampy wetlands cover much of the Property along with mixed forest composed mainly of spruce, pine grey, balsam fir, poplar and birch. The surrounding land has an altitude of about 290 to 315 masl.

6. HISTORY

This section was adapted and translated from the 2019 NI 43-101 technical report (Bonneville, 2019), which describes the exploration and development history of the Project and reviews recent transactions and agreements between companies regarding the Project. Table 6.1 summarizes major exploration phases on the Property since its discovery.

Table 6.1 – Historical gold production from the Sleeping Giant Mine

Period	Company	Tonnes	Grade (g/t Au)	Contained ounces
1988 to 1991	Perron Gold / Aurizon Mines	494,000	6.10	96,883
1993 to 2008	Cambior / IAMGOLD	2,633,200	11.40	965,116
2009 to 2012	Cadiscor / NAP	206,640	6.01	39,928
2013 to 2014	Aurbec	33,251	5.71	6,104
Total		3,367,091	10.24	1,108,031

6.1 Perron Gold Mines / Aurizon Mines

In 1983, Perron Gold Mines (later Aurizon Mines Ltd; “Aurizon”) acquired a 50% interest in the Property by conducting ground-based geochemical, magnetic and very low frequency (“VLF”) surveys, diamond drilling and underground exploration work. Between 1984 and 1987, two (2) shafts were sunk, and sufficient historical mineral reserves were delineated to begin development work. The first phase of commercial production lasted from 1988 to 1991, during which 494,000 t at 6.1 g/t Au were extracted from levels 55 to 415. At the end of 1991, Aurizon Mines, then the sole owner of the Sleeping Giant mine, ceased operations due to the depletion of the historical reserves and a severely depressed gold price.

6.2 Cambior / IAMGOLD

In 1991, an agreement between Aurizon and Cambior Inc. (“Cambior”) allowed Cambior to acquire a 50% interest in the Property by investing in underground drilling and development. Under the terms of this agreement, Cambior was the project manager. Between 1991 and 1993, approximately 13,354 m of drilling led to the discovery of four new mineralized zones (20, 30, 40 and J-D) and the start of the second phase of commercial production that lasted from 1993 to 2008. In 1993, Cambior conducted 16 reverse circulation (RC) holes at Sleeping Giant, followed by a reconnaissance program of 152 RC holes on its nearby Dormex property. The key highlights of this period included the discovery of zones 2, 3, 4, 5, 6, 7, 8, 9, 16, 18 and 50, and the sinking of the main shaft in two stages: levels 485 to 785 in 1995 and levels 785 to 975 in 2003. In November 2006, Cambior’s assets were acquired by IAMGOLD Corporation.

On October 9, 2007, IAMGOLD signed an agreement with Cadiscor Resources Inc. ("Cadiscor") granting them the right to purchase the Sleeping Giant Property upon completion of operation and mill processing of existing historical reserves. Mining operations ceased in the summer of 2008, and the mill extracted its last ounces of gold in October 2008. From 1993 to 2008, a total of 962,300 ounces of gold were mined from 2,633,200 t of ore at an average grade of 11.4 g/t Au.

6.3 Cadiscor / North American Palladium

After the transaction, Cadiscor completed a program of 90 underground drill holes totalling 18,669 m. The drilling led to a historical mineral reserve estimate by Birkett et al. (2008).

In May 2009, North American Palladium Ltd ("NAP") acquired Cadiscor's shares, becoming the sole owner of the Sleeping Giant mine and mill along with the Dormex, Flordin and Discovery gold projects. In October 2009, NAP restarted production at the Sleeping Giant mine and purchased the 1% royalty held by IAMGOLD for $1 million. During the same year, NAP completed 24,718 m of definition drilling, 11,017 m of exploration drilling and 21 RC holes. In 2010, NAP began deepening the main shaft to allow for the development of levels 1060, 1115 and 1175. That same year, 49,402 m of definition drilling and an additional 37,862 m of exploration drilling were completed. Finally, a program of 113 RC holes was conducted on the Sleeping Giant and Dormex properties (MacNeil, 2011). Another historical mineral reserve estimate was released by Jourdain et al. (2010) based on those new drill holes. In March 2011, the 2010 historical reserves were updated by Jourdain et al. (2011).

Following a drilling program of 116 drill holes totalling 36,746 m, NAP indicated that although the gold zones extend at depth, the structures mined to date are discontinuous, significantly reducing the tonnage that could be profitably mined. As a result, NAP suspended its mining activities at the mine in early 2012 (SEDAR: NAP, January 17, 2012). From 2009 to 2012, NAP processed 206,640 t at a grade of 6.01 g/t Au for 37,328 ounces of gold.

6.4 Aurbec Mines

Under an agreement signed on March 22, 2013, Maudore Minerals Ltd ("Maudore") acquired Sleeping Giant's mill and NAP's gold assets in Quebec. Maudore subsequently created a subsidiary, Aurbec Mines Inc. ("Aurbec"), which took control of the new properties. On August 16, 2013, Aurbec signed an agreement with Abcourt to process gold from the Elder mine for six months. A few days later, Aurbec announced the start of an underground exploration drilling program to target the extensions of known zones (3-8-20-30) and explore new mineralized structures, including the 785N vein. When the program wrapped up in June 2014, Aurbec had completed 167 exploration and definition drill holes totalling 26,781 m. The company also continued underground development and ore extraction from existing stopes.

On October 29, 2013, InnovExplo Inc. completed a historical mineral resource estimate and technical report (Verschelden and Jourdain (2013).

On November 7, 2014, Maudore announced the cessation of operations at the mine, followed two months later by the bankruptcy of its subsidiary Aurbec. Maudore also transferred its assets on May 16, 2016. From 2013 to 2014, Aurbec processed 33,251 t of ore at a grade of 5.71 g/t Au.

6.5 Abcourt Mines

On June 20, 2016, following Aurbec's bankruptcy, Abcourt acquired all the sequestered mining assets from Deloitte for a cash sum of $2,548,727 and a royalty of $5.00 per tonne extracted from the Property, up to a maximum of 350,000 t. The mine has been on care and maintenance since then, with only underground diamond drilling activities.

Abcourt has kept the mill functioning since the acquisition by processing ore from its Elder mine in Rouyn-Noranda. This came to an end in 2022.

6.6 Summary of Historical Gold Production

Past owners have completed several phases of exploration and definition work, historical mineral resource and reserve estimates, and several production phases (Table 6.2).

Table 6.2 – Selected historical work on the Property

Year	Company	Work description	Details	References
1948-1949	Ministère des Ressources Naturelles	Geological mapping	A few geological targets were highlighted. Quartz-carbonate veins mineralized in pyrite and chalcopyrite in an area at the western end of the Property.	Tiphane,1948; Tiphane,1949
1956-1957	Harricana Prospection Syndicate	Prospecting; airborne EM and Mag survey	Covered some parts in the west and north parts of the Property. Good correlation between the two surveys ground geophysics follow-up on several coincident anomalies.	Forbes and True, 1957; Forbes, 1957; Smellie, 1957
1957	American Metal Climax Inc.	Airborne EM survey	Western part of the Property; many conductors identified.	Ward, 1958
1958-1959	Ministère des Ressources Naturelles	Geological mapping	Entire Sleeping Giant Property covered	Imbault and Remick, 1958; Remick, 1964; Tiphane, 1959
1958-1959	Kennco Explorations Ltd.	Airborne EM, TURAM and VLF surveys Mapping, geochemistry	NE and NW parts of the Property	Black, 1958; Black, 1959
1961	Quebec Mattagami Minerals Ltd.	Airborne geophysical survey	Two parts at the northern boundary of the Property were covered. Survey did not detect any good sulphide prospects but showed that the amount of peridotite might be quite significant.	Lang and Prendergast, 1961
1963-1964	Rio Tinto Canadian Exploration Ltd.	Ground geophysical surveys (Mag, EM, gravimetry) 4 surface DDH (1 to 4) for 800 m	NW corner of the Property. DDH #4 was located on the property; peridotite with some disseminated pyrite and asbestos stringers was identified, but no assays were taken.	Boniwell, 1963; Agar et al., 1964
1966-1974	Abitibi Asbestos Mining Company Limited	Mag survey 7 surface DDH for 1,396 m (not on the Property) EM survey, FS report, bulk sampling, drill core logging (B-1-1-72 459 m), preliminary study on slope angles 4 surface DDH (74-C-1 and 2; B-6-78 and B-2-78) for 1,081 m on the B and C	Survey covered the NW end part of the Property; discovery of a large asbestos-bearing deposit. DDH 74-C-1 and 74-C-2 on the Sleeping Giant Property outlined significant asbestos mineralization associated with altered peridotite	Ingham, 1967; Ingham, 1968; Foy et al., 1971 Foy and Ingham, 1972; McCammon et al., 1974; Osborne, 1974; Osborne and Descarreaux, 1978

Year	Company	Work description	Details	References
		properties		
1966	Lansdowne Explorations Limited	Ground Mag and EM survey 13 surface DDH (LS-1 to LS-11) for 1,636 m	10 of the 13 DDH were drilled in part of the NNW corner of the Property	Pudifin, 1966
1976	Mattagami Lake Exploration Ltd	Questor Airbourne INPUT survey 4 surfaces DDH (SM-76-2 to 5) for 642 m	Mattagami Lake commissioned as part of a regional zinc exploration program Best result: 8.3 g/t Au / 1.1 m (SM-76-5)	King and Sullivan, 1977; Huertas and Chevalier, 1984; King, 1976
1977-1980	Mattagami Lake Exploration Ltd	IP, Mag and EM surveys 2 surface DDH (SM-77-13-14) for 705 m 8 surface DDH (SM-78-15 to 22) for 1,807 m 12 surface DDH (SM-79-25 to 29, 34a, SM-80-30 to 33, 33a and 17ext) for 2,269 m	IP survey conducted on a detailed grid	Halliwell and Sullivan, 1977; Sullivan, 1978; Sutherland, 1979; Huertas and Chevalier, 1984; Descarreaux, 1983
1981	Mattagami Lake Exploration Ltd	IP survey 8 exploration DDH: SM-81-50A, 50B to 56 inclusive., and SM-81-72 (1512.46 m) 32 development DDH: SM-81-35 to 49 inclusive, SM-81-57 to 71 inclusive, SM-81-73 and 74 (5726.59 m)	Reconnaissance grid, Group M West Extension and Group M (north and east of the 1977 survey) Exploration holes tested IP anomalies in zones A, D, E, G, I and J Development holes: definition drilling on the A vein	Huska, 1981; Sutherland, 1981; Huertas and Chevalier, 1984; Huertas, 1984
1982	Mattagami Lake Exploration Ltd	Completion of a Pre-Evaluation study and tonnage calculation on the A vein.	Total tonnage for the A vein estimated at 526,278 short tons (Indicated, Possible and Probable) with a grade of 4.99 g/t Au and 12.26 g/t Ag based on a minimum mining width of 2.0 m.	Descarreaux, 1983
1983	Mattagami Lake Exploration Ltd Perron Gold Mines	Mattagami Lake Exploration and Perron Gold Mines sign an agreement	Acquired a 50% interest in the Property	Huertas and Chevalier, 1984; Descarreaux, 1983
1984-1987	Perron Gold Mines	EM and Mag surveys 54 DDH for 13,469 m Re-logging of all previous drill core	Sufficient reserves were delineated to begin development work	Smith and Huertas, 1987; Husson, 1984; Huertas, 1984;

Year	Company	Work description	Details	References
		2 shafts sunk in 1986-1987 Underground excavations (drifts and crosscuts) 45 DDH for 9,517 m FS in 1987		Lecuyer, 1987; Ross, 1985; Mayer, 1987
1986	Placer Development Ltd.	4 surface DDH (GL86-001 to 004) for 1,023.3 m	Assay results fail to outline any one prime area for future drilling	Kowalczyk and Andresen, 1986; Andresen, 1986
1986	Imco Resources Inc.	Airborne Mag-EM survey	EM survey shows essentially the same features as the last government survey	Smith, 1987
1988	Noranda Mines Ltd Perron Gold Mines (Mines Aurizon)	5 surface DDH (408-3, 411-1, 411-5 to 7) for 2,168 m on the Faith/Perron Property in the middle of the Sleeping Giant Property 4 surface DDH (88-1 to 4) for 620 m on the Chaste Property in the NE part of the Sleeping Giant Property	Best results: 91.7 g/t Au / 0.5 m (hole 408-3) 48.4 g/t Au / 0.6 m (hole 411-7) No significant results in Hole 88-1	Coates, 1988
1988-1991	Noranda Mines Ltd Perron Gold Mines (Mines Aurizon)	First phase of commercial production	494,000 t at 6.1 g/t Au extracted from levels 55 to 415	Charlton, 1989
1991	Mines Aurizon	Sole owner of the Sleeping Giant mine	Operations cease due to the depletion of reserves and a severely depressed gold price	Houle, 2001; GM 51261
1991	Mines Aurizon Cambior	UG drilling and UG development	Agreement between Aurizon and Cambior allows Cambior to acquire a 50% interest in the Property; Cambior is project manager	Desjardins et al., 1992
1991-2006	Mines Aurizon Cambior IAMGOLD Corporation	Drilling (13,354 m): 16 RC holes at Sleeping Giant Reconnaissance program of 152 RC holes on the Dormex Property IP and ground geophysics	Discovery of four new mineralized zones (20, 30, 40 and J-D) Discovery of lenses 2, 3, 4, 5, 6, 7, 8, 9, 16, 18 and 50 Sinking of the main shaft in two stages: levels 485 to 785 in 1995 and levels 785 to 975 in 2003	Averill, 1983; Desjardins, 1994 Plante, 1993; Gobeil, 1995; Gilbert, 1995; Gauthier, 1998; Berube, 1998; Plante, 1998; Berube, 1999; Ducharme, 1999; Lambert, 2004; Hubert,

Year	Company	Work description	Details	References
			IAMGOLD acquires Cambior in 2006	2005; Villeneuve and Caron, 2005; Villeneuve, 2006; Birkett et al., 2008
2007-2008	IAMGOLD Corporation Cadiscor Resources Inc.	IAMGOLD signs agreement with Cadiscor granting them the right to purchase the Property upon completion of mining operations and milling of existing reserves. 90 UG DDH totalling 18,669 m	Mining operations cease in the summer of 2008 Mill extracts last ounces of gold in October 2008 1993 to 2008: 962,300 ounces of gold mined from 2,633,200 t of ore at an average grade of 11.4 g/t Au Historical mineral reserve estimate prepared	Birkett et al., 2008
2009-2011	North American Palladium Ltd NAP Quebec Mines Ltd.	Sleeping Giant mine: Production restart 24,718 m of definition drilling 11,017 m of exploration drilling 21 RC holes Deepening of the main shaft Development of levels 1060, 1115 and 1175 New mineral reserve estimate (update of 2010 estimate using 116 holes totalling 36,746 m) Sleeping Giant and Dormex properties: 49,402 m of definition drilling 37,862 m of exploration drilling 113 RC holes	Acquisition of Cadiscor Resources Inc. completed 1% royalty held by IAMGOLD purchased for $1 million Gold zones extend at depth; structures mined to date are discontinuous, significantly reducing the tonnage that could be profitably mined Mining activities suspended in 2012 From 2009 to 2012, 206,640 t processed at a grade of 6.01 g/t Au or 37,328 ounces of gold	MacNeil, 2011; Jourdain et al., 2010; Jourdain, et al., 2011; NAP, January 17, 2012; Lussier and Birkett, 2013
2013-2014	Maudore Minerals Ltd Aurbec Mines Inc.	Acquisition, creation of subsidiary 167 exploration and definition holes for 26,781 m Continuation of UG development work	Acquisition of NAP's gold assets in Quebec and creation of a subsidiary, Aurbec Abcourt to process gold from the Elder mine for 6 months Historical mineral resource estimate prepared for Sleeping Giant deposit	Jourdain and Verschelden, 2013

Year	Company	Work description	Details	References
		and mining of existing stopes	Cessation of mining operations followed 2 months later by bankruptcy of subsidiary From 2013 to 2014, Aurbec processes 33,251 t of ore at a grade of 5.71 g/t Au Maudore transfers its Aurbec assets on May 16, 2016	

7. GEOLOGICAL SETTING AND MINERALIZATION

The sections below on the regional geology, mine geology and mineralization are adapted from a PhD thesis on the Sleeping Giant gold deposit and its potential volcanogenic origin (Gaboury, 1999).

7.1 Regional Geology

The Sleeping Giant deposit is hosted within the first volcanic cycle of the Northern Volcanic Zone of the Archean Abitibi Subprovince (Chown et al., 1992) (Figure 7.1). The cycle corresponds to an extensive subaqueous basalt plain with scattered felsic volcanic edifices, interstratified with or overlain by volcaniclastic assemblages. The 1 to 3 km thick basaltic assemblage (Chown et al., 1992) is dominantly composed of tholeiitic massive, pillowed and breccia volcanic rocks (Picard and Piboule, 1986), with minor chert, iron formation and volcaniclastic rocks (Mueller and Donaldson, 1992; Pilote, 1989). U-Pb dating of felsic centres in the middle to the upper part of the first volcanic cycle indicates a time interval of 2730-2720 Ma (Mortensen, 1993). The Northern Volcanic Zone is interpreted as a coherent assemblage, initially formed from a diffuse and immature arc (Chown et al.,1992).

The Northern Volcanic Zone was affected by a N-S shortening event from 2708 to 2685 Ma; this comprised a succession of several tectonic pulses of continuous deformation (Chown et al., 1992). The main deformational features are (1) E-W trending, subvertical, regional folds with an axial planar fabric, (2) major E-W trending, 1 to 4 km wide reverse deformation zones of regional extent, and (3) dextral, 1 to 5 km wide, NW-SE trending deformation zones.

At the district scale, three voluminous synvolcanic and polyphase (diorite-tonalite leuco-tonalite) plutons are responsible for the disturbed regional structural trend (Chown et al., 1992) (Figure 7.2). The Sleeping Giant mine is located close to the centre of a triple junction zone of the structural trend (Daigneault and Archambault, 1990). Other important features of the area include the E-W trending Laflamme fault and the NW-SE trending Harricana fault (Figure 7.3).

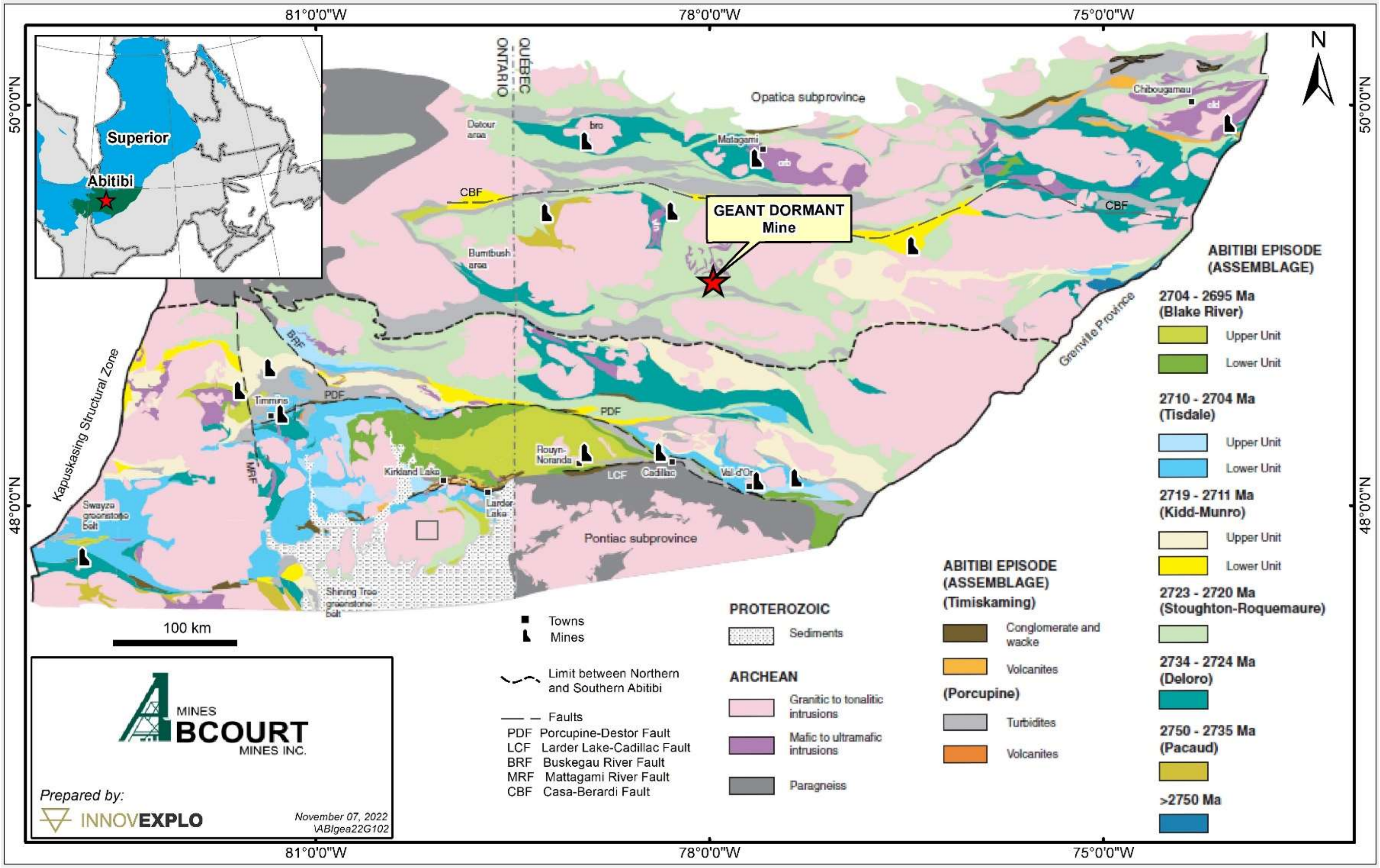
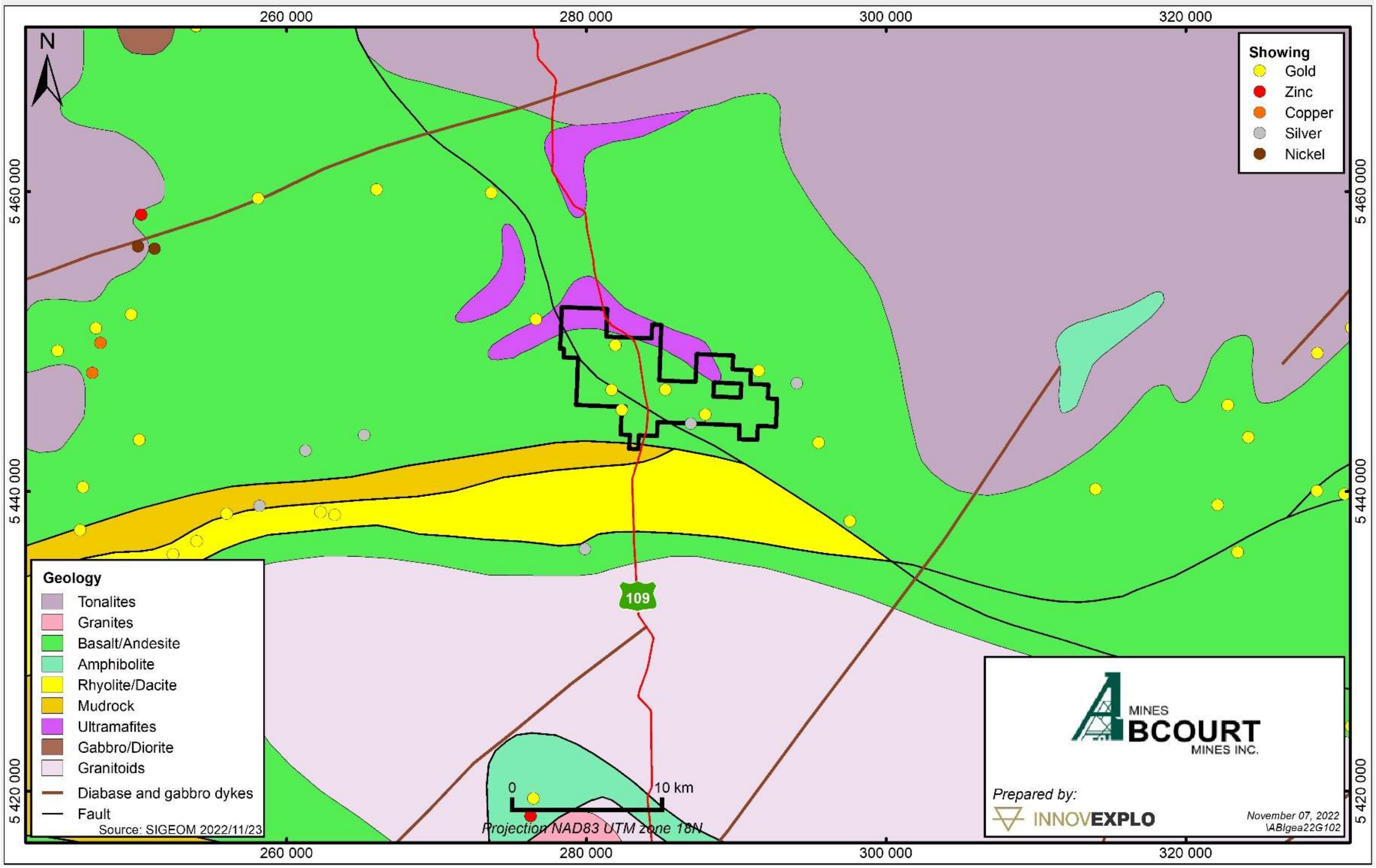


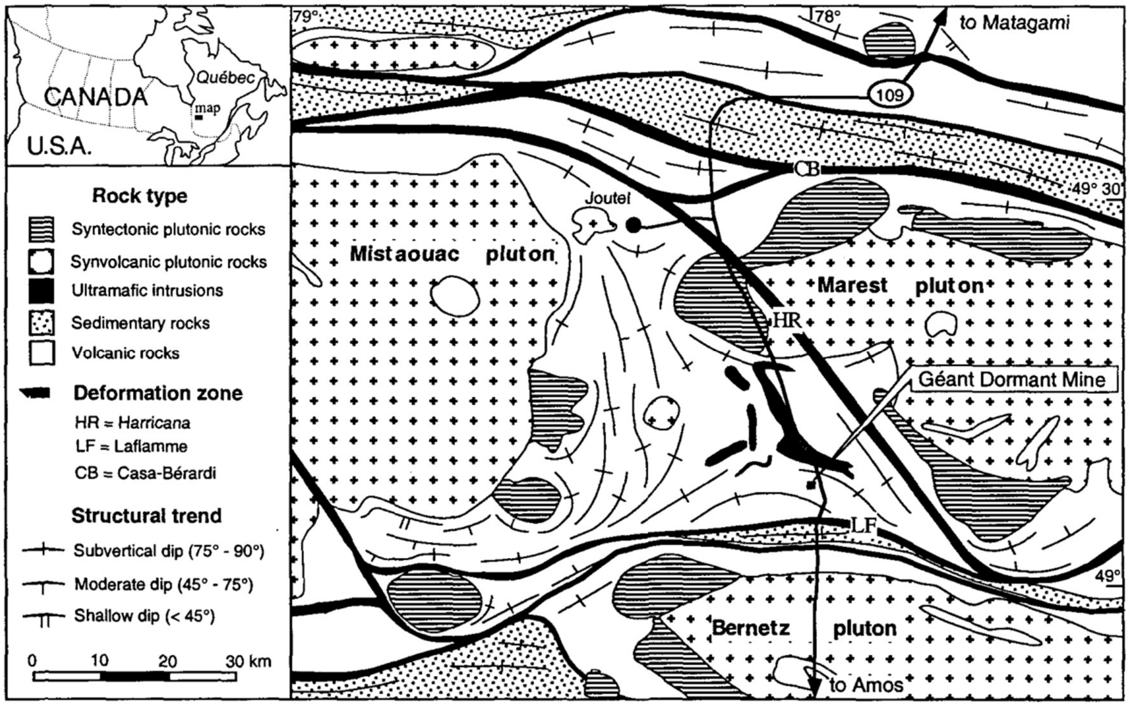
Figure 7.1 – Regional Geology – Abitibi Subprovince



Source: SIGEOM 2022/11/23

Projection NAD83 UTM zone 18N

Figure 7.2 – Local Geology



showing the location of the Sleeping Giant mine close to the central part of a triple junction of structural trends. Map modified from Hocq (1990) and Chown et al. (1992).

Figure 7.3 – Simplified geological map of the mine district

7.2 Local Geology

The tectonostratigraphy of the area comprises two volcanic cycles (North Cycle and Mine Cycle) in relation to a large intrusive complex (Figure 7.4). At the base of the stratigraphic sequence is the Northern Cycle (northwest part of the Property), which contains mainly iron-rich tholeiitic basalts and co-magmatic sills of gabbro. These tholeiites are easily distinguished from the tholeiites of the Mine Cycle by their higher TiO_2 content (>1%). Stratigraphically above and concordant with the Northern Cycle, the Mine Cycle sequence represents the dominant host sequence of Sleeping Giant. This cycle contains mainly magnesium-rich tholeiitic basalts and co-magmatic sills of gabbro. Laminar deposits composed of fine sediments, tuffs and iron formations (with magnetite) are inter-stratified in the sequence. The sedimentary and volcaniclastic rocks define larger amalgamated units in the central part of the Mine Cycle.

The stratigraphic sequence of the Mine Cycle is intersected by intrusions of intermediate to felsic composition and calc-alkaline affinity that constitutes the Sleeping Giant Complex and are penecontemporaneous with the Mine Cycle volcanism. The Sleeping Giant Complex includes a main dacitic mass, several satellite dacitic units and a swarm of felsic porphyry dykes.

Four main phases are recognized in the magmatic evolution of the complex:

1. dacite with mafic phenocrysts (chlorite spots);
2. dacite with feldspar phenocrysts of and mafic feldspar.
3. porphyry with quartz+feldspar phenocrysts (with granitoid texture locally);
4. porphyry with quartz phenocrysts.

The main dacitic mass occupies the central part of the mine and is up to 400 m thick. The geochemical signature of this dacite is typical of a family of similar gold deposits in the area, including the Bousquet mining camp (Gaboury and Pearson, 2008). The other intrusive phases intersect the entire volcano-sedimentary sequence, as well as the main dacitic mass, at high angles and are oriented NW-SE to WNW-ESE. These vary in width from centimetric- to metric-scale with the thickest occasionally showing evidence of polyphase injections.

Some post-mineralization tholeiitic dykes have been observed. These predate the main deformation. As several of them parallel gold-bearing veins they have been used to target extensions of gold-bearing structures.

A quartz porphyry mass in the SW area and a NE-SW oriented sericite unit represents late felsic intrusions relative to the main deformation phase. These intrusions are geochemically similar to each other and are differentiated from those of the Sleeping Giant Complex by their low MgO and TiO_2 contents, and their low (i.e., <5) Zr/Y ratios.

A lamprophyre with hornblende phenocrysts (the “gabbro” in mine nomenclature) crosses the entire mine sequence. It is 5 to 25 m thick, oriented NW-SE and has a shallow dip to the NE. The lamprophyre is late relative to the main deformation phase. Several minor dykes of similar type are observed at various locations in the stratigraphic sequence.

Recent data indicate that the mine area is the site of a tight synform with an axial-plane oriented ESE-WNW (Figure 7.4). The general plunge of the fold is moderate to the east. In the northern sector of the mine, seams are oriented ESE-WNW with a steep dip to the south. In the western sector these layers strike NNE-SSW with a moderate dip to the east. They return to SE-NW in the southern sector. The structure is interpreted as a box fold (Figure 7.6).

NW-SE faults mark another important structural aspect of the mine area. Two categories of NW-SE faults are distinguished in the mine environment:

1. ductile with dextral component (SW area)
2. brittle with a sinistral component (NE area)

Both categories of faults are late in the deformational history and displace the mineralized zones. In the SW sector of the mine, drilling revealed the presence of a large NW-SE ductile dextral fault zone focused mainly within a shale interval. This fault zone dips about 70° to the NE and has a horizontal offset of approximately 2 km. The characteristics suggest that this fault belongs to the family of NW-SE dextral strike-slip faults that is recognized throughout the Abitibi Subprovince. Typically, favourable lithologies south of this fault are barren of economic mineralization. A prominent NW-SE brittle sinistral fault occurs in the NE part of the mine area.

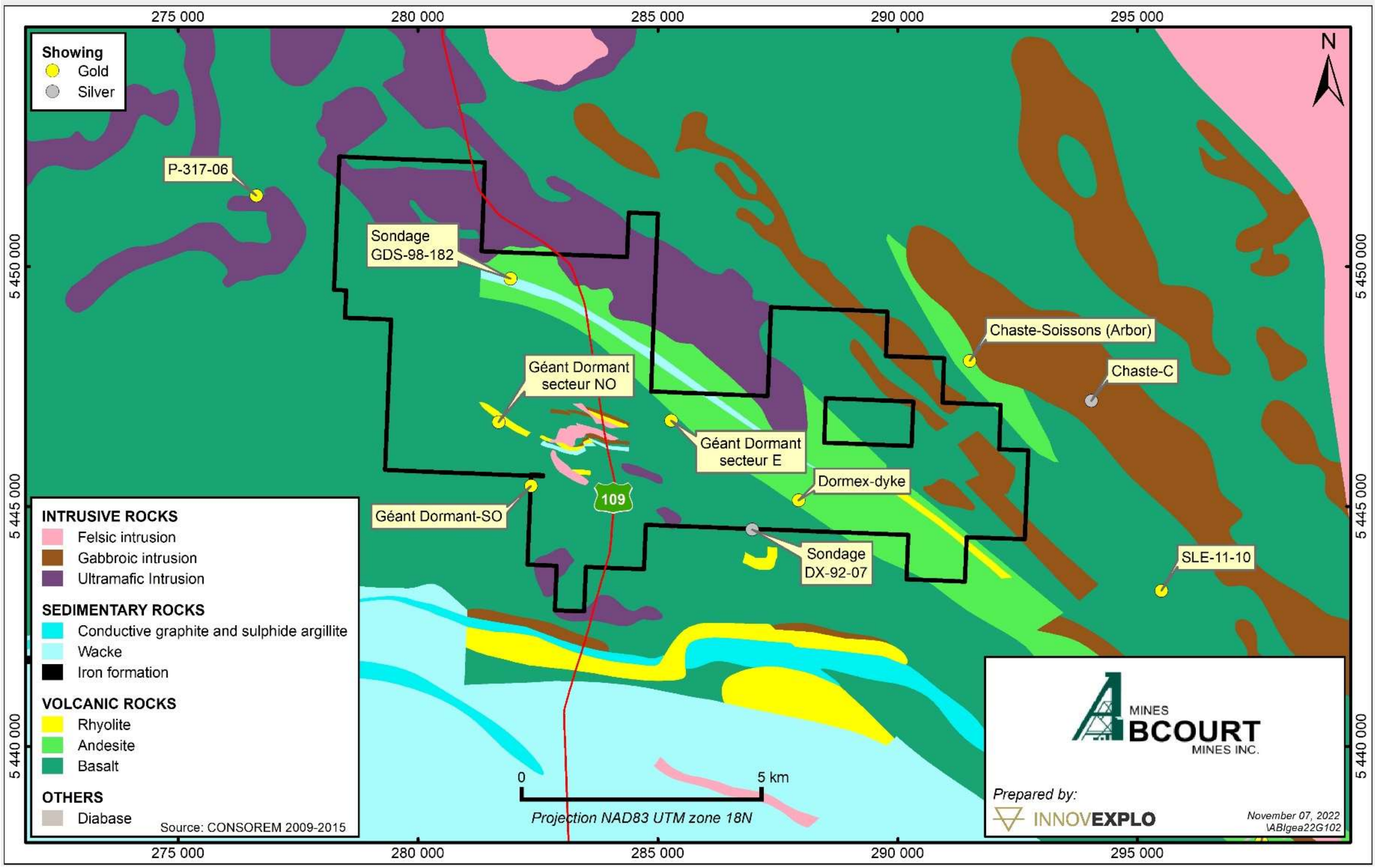
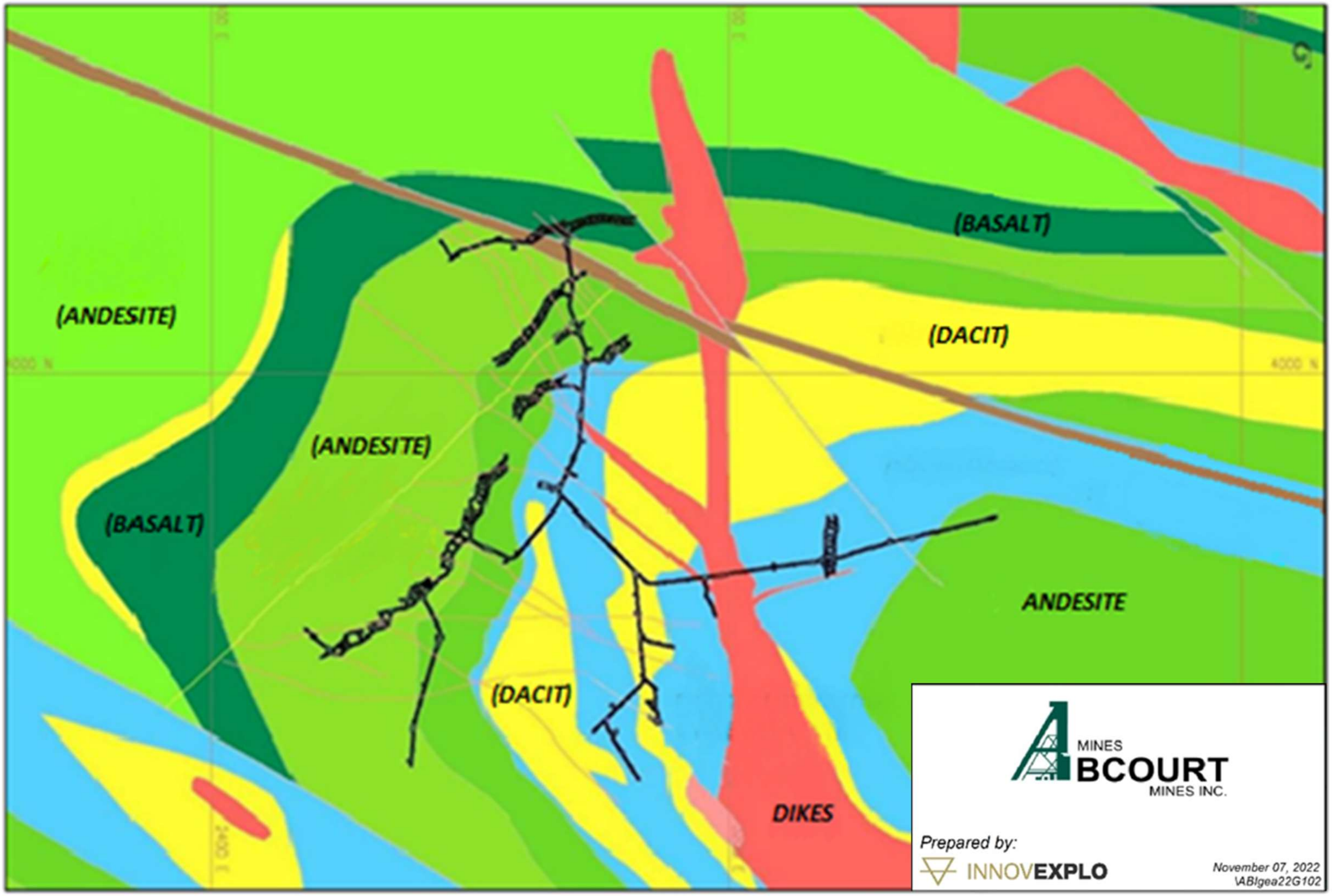
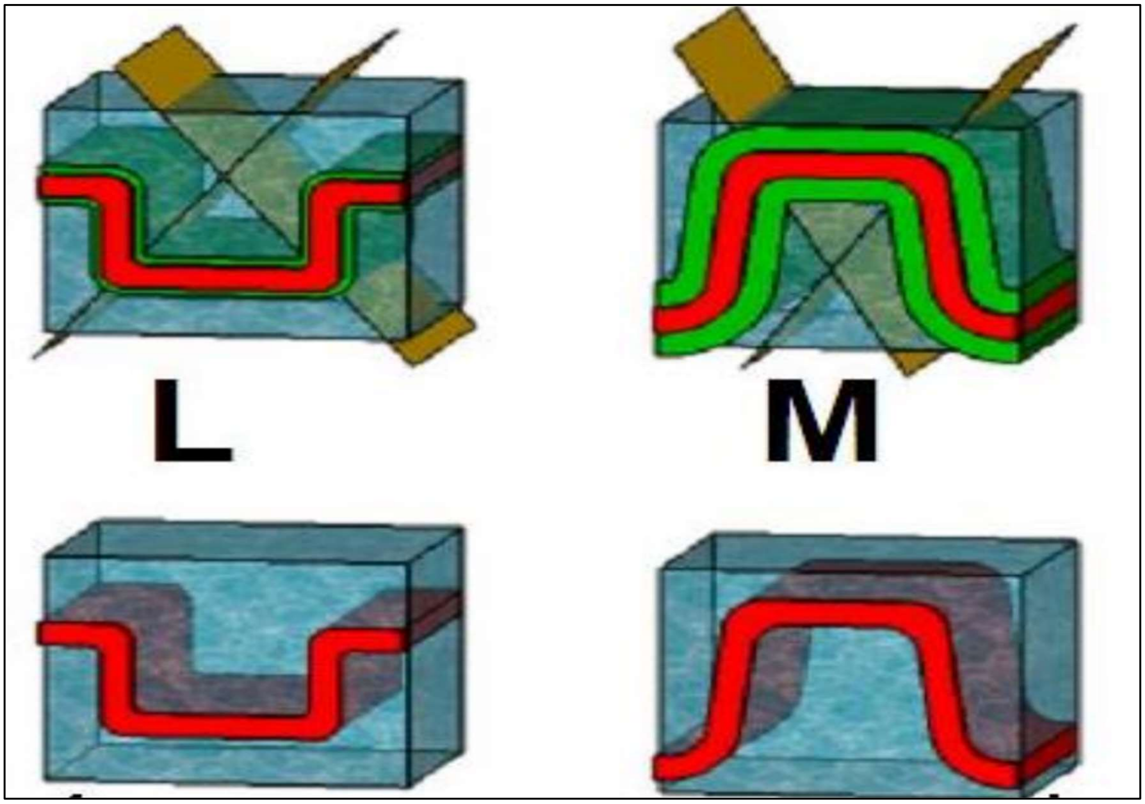


Figure 7.4 – Geological base map of the Property



Underground development of level 665 superimposed.

Figure 7.5 – Geological map showing the hinge area of the east -plunging syncline in the mine vicinity.



(Source: http://www.geologues-prospecteurs.fr/documents/plis/)

Figure 7.6 – Schematic examples of box folds

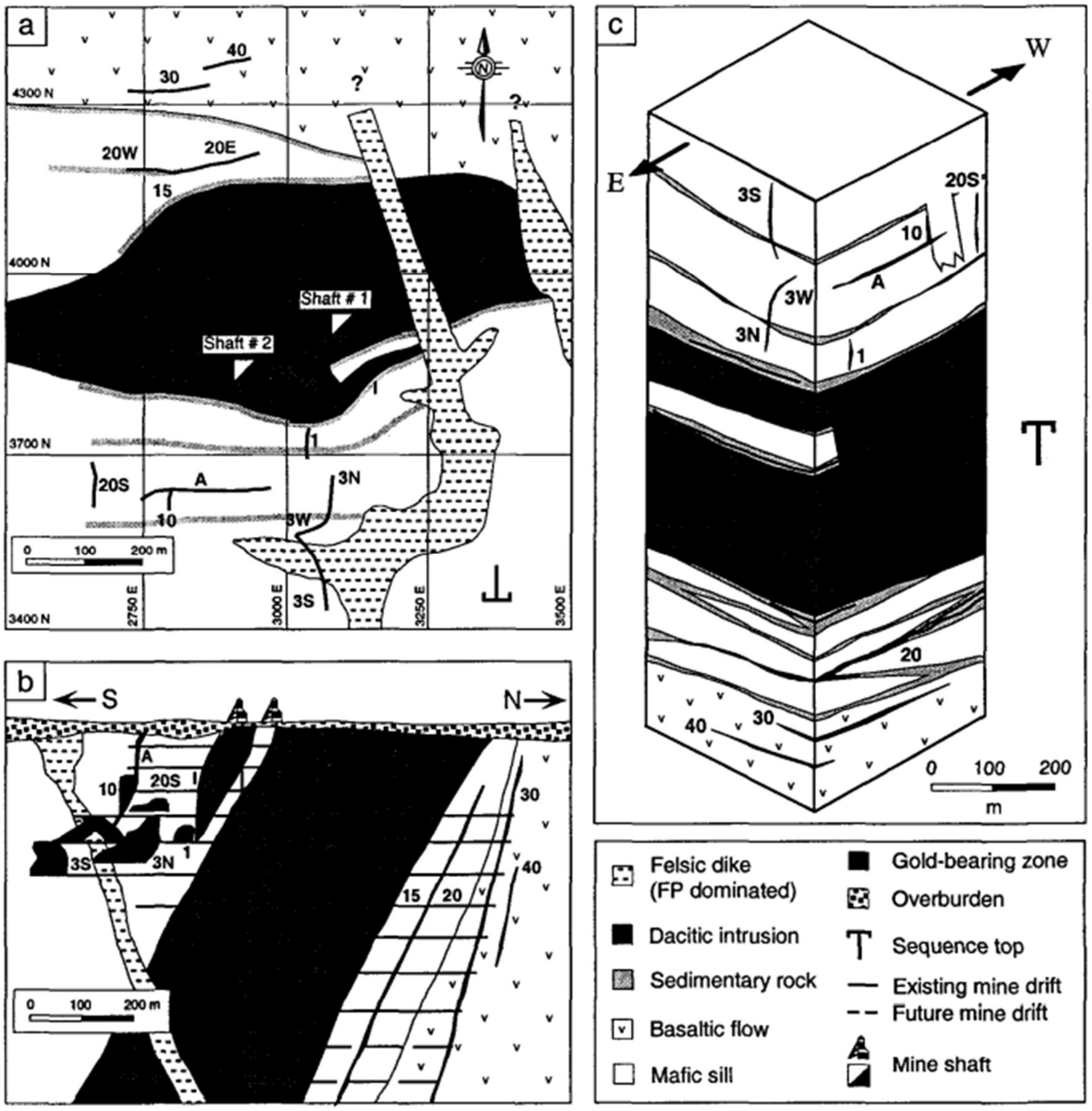
This fault dips about 65° to the NE and shows a horizontal offset of approximately 500 to 1,000 m. There are similar NW-SE faults that intersect with the main dacitic mass and associated with the lamprophyre dyke. The latter runs along the dyke and is slightly inclined towards the NE with an offset of about 100 m.

7.3 Mine Geology

The mine sequence is represented by a volcano-sedimentary succession intruded by a felsic complex and transected by late hornblende-rich dykes. Strata strike predominantly E-W and dip steeply to the south, forming a single homoclinal stratigraphic succession (Figure 7.7). All the rock types have been metamorphosed to greenschist facies, but the prefix "meta" is omitted in the following sections to simplify the rock nomenclatures.

7.3.1 Volcano-sedimentary succession

The volcano-sedimentary succession comprises, from north to south, a basaltic flow unit overlain by sedimentary rocks with a swarm of intercalated mafic sills. The basaltic flow, the lowest stratigraphic unit exposed within the mine (Figure 7.7), exhibits a southward succession of massive, pillowed and pillow breccia, which, together with well-preserved pillow configurations, indicate younging to the south. Sedimentary horizons enclosed between two mafic sills range in thickness from a few centimetres to 10 m, with an average of 2 m. Sedimentary rocks consist of fine-grained volcaniclastic, chert and oxidesulphide facies iron formation. Local flame textures and graded bedding indicate southward younging of the sedimentary succession. Mafic sills, 1 to 100 m in thickness, are a ubiquitous feature in the mine environment. They are aphanitic to medium-grained and locally plagioclase-phyric. South of the dacitic body (Figure 7.7), four main mafic sills of great lateral and vertical continuity are well delineated, whereas the mafic sill swarm is more heterogeneous in the northern part (Figure 7.7a and Figure 7.7c).



(a) Simplified geological plan of the Sleeping Giant mine area, (b) Idealized vertical N-S cross section, (c) Schematic stratigraphic block. Various gold-bearing zones are labelled: A, 10, 40, etc. Figures a and b modified form maps of Cambior Inc.

Figure 7.7 – Characteristics of the mine sequence

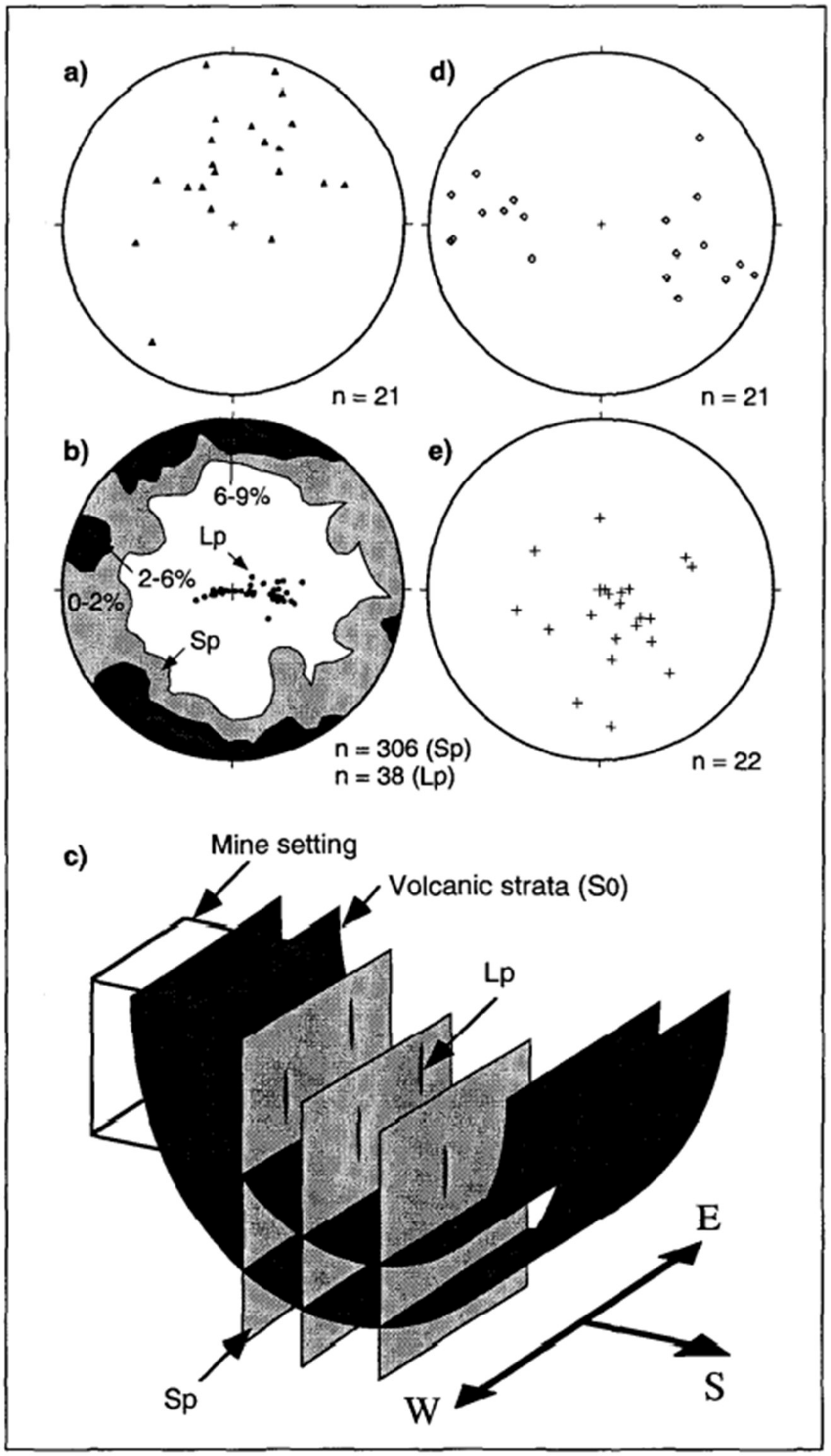
The mafic rocks have a tholeiitic geochemical signature (Gaboury et al., 1998), which is typical of the first volcanic cycle (Picard and Piboule, 1986).

7.3.2 Felsic intrusive complex

The central dacitic body (Figure 7.8) and a swarm of felsic porphyry dykes constitute the felsic intrusive complex. The dacite is aphanitic and homogeneous and forms a 400 m wide lenticular body injected subconformably into the mafic sill swarm (Figure 7.8). Felsic porphyry dykes crosscut both the volcano-sedimentary succession and the dacitic intrusion. Dykes are divided into two main groups on the basis of phenocryst contents: the plagioclase-phyric ("FP") and quartz and plagioclase-phyric ("QFP"). The QFP dykes systematically crosscut the FP dykes and form tabular injections, whereas the latter are more irregular. The QFP dykes are oriented mainly SE-NW and dip steeply to the NE. Individual dykes are typically less that 4 m wide; wider dykes are generally composite intrusions of FP and QFP. The dacite and felsic dykes have calc-alkaline geochemical signatures (Gaboury et al., 1998), which are comparable with those of synvolcanic felsic rocks of group FP (Lesher et al., 1986) and group II (Barrie et al., 1993) from the Abitibi Subprovince.

7.3.3 Late hornblende-rich dykes

Fine- to coarsely-crystalline mafic dykes, containing 40 to 60% euhedral hornblende phenocrysts, are ubiquitous throughout the mine. These dykes (< 5 m wide) cut all the other rock types, and the gold-bearing mineralized lenses. They display variable attitudes (Figure 7.8a), well-developed chilled margins and step-shaped contacts. The shoshonitic geochemical signature of these dykes (Gaboury et al., 1998) is similar to that of Archean lamprophyres (Wyman and Kerrich, 1993) and late shoshonitic volcanic rocks of the Chibougamau area (Dostal and Mueller, 1992).



(a) Equal area projections (lower hemisphere) of Sp planar fabric and Lp stretching lineation. (b) Schematic structural setting of the deposit Mineralization

Figure 7.8 – Structural features of the deposit

7.3.4 Mineralization styles

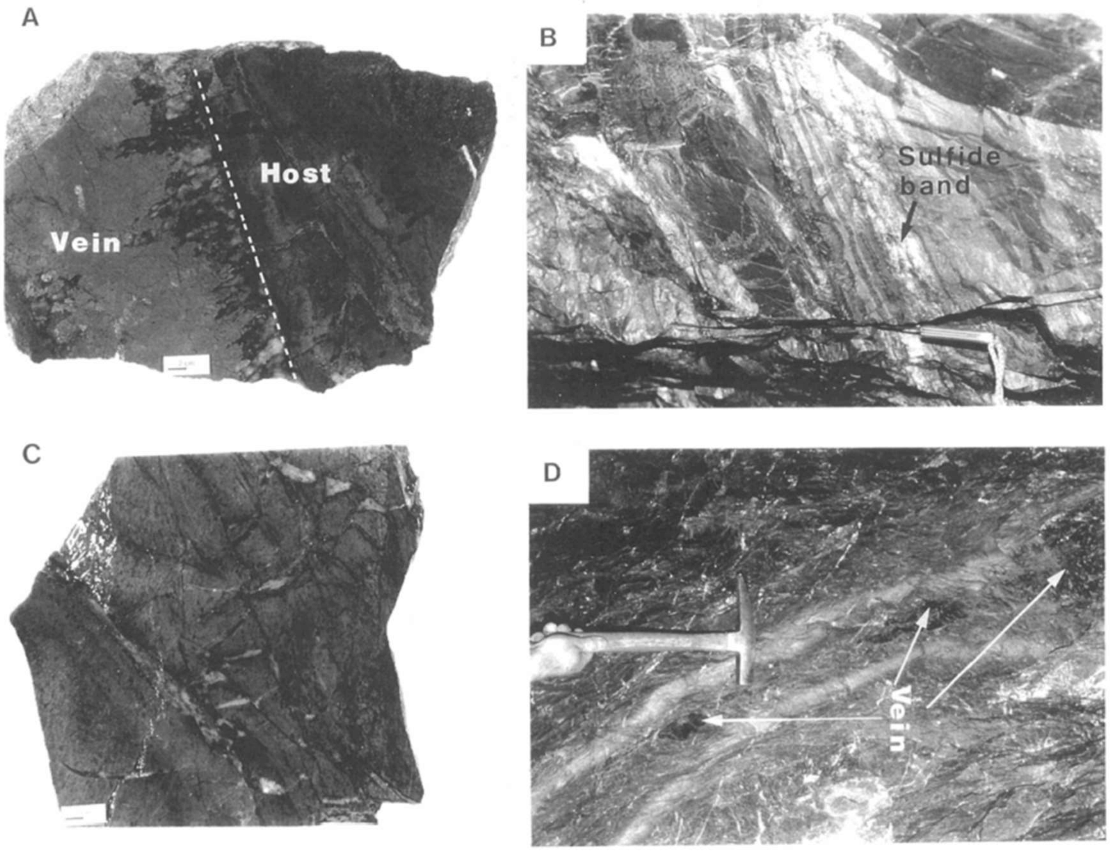
The sections below describe the following styles of gold mineralization found in the Sleeping Giant mine: (1) gold-bearing veins (economically the most important), (2) stratabound gold-bearing, (3) SE-NW veinlet arrays, and (4) veining within the QFP dykes. The last two styles are uneconomic, but they are significant in establishing the controls and timing of gold mineralization.

7.3.4.1 Gold-bearing veins

Gold-sulphide-quartz veins are generally massive and range from a few centimetres to 2 m thick (average of 50 cm for the mined veins). The veins are rich in gold with assays commonly > 100 g/t Au (unpublished data, Cambior Inc.). The vein contacts range from sharp, planar and free of surrounding planar fabric (Figure 7-9a) to undulose and schistose. Branching of main veins is a common phenomenon but does not show consistent attitudes. Veins terminate laterally through pinching out or arborescent multi-branching veinlets (Figure 7.10b).

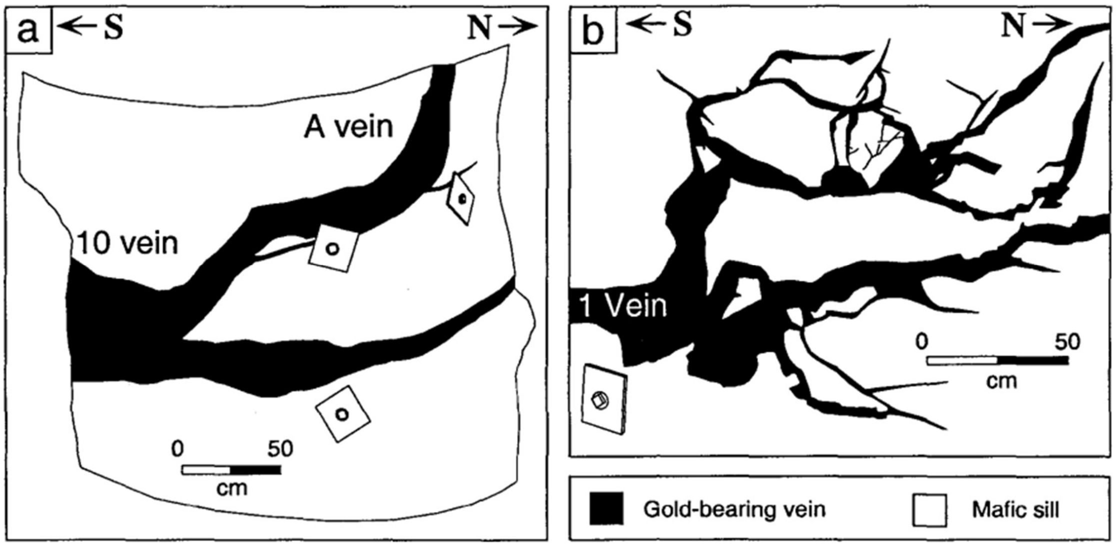
7.3.4.2 Stratabound mineralization

Stratabound mineralization corresponds to gold-bearing segments of sedimentary horizons in which three main modes of occurrence have been observed: (1) subconcordant, millimetre- to centimetre-wide quartz-sulphide veinlets (the most common form); (2) concordant, parallel layers of massive sulphides; and, (3) disseminated sulphides, mainly pyrrhotite, pyrite and marcasite, with minor chalcopyrite and sphalerite. The sulphide layers are a few millimetres thick and are mainly composed of pyrrhotite and pyrite, with lesser chalcopyrite and sphalerite (Figure 7.9b). In thin section, remnant magnetite grains and pseudomorphs suggest that the sulphide layers, and in part the disseminated sulphides, formed by the hydrothermal replacement of magnetite iron formations. Mineralization occurrences are related to both compositional (e.g., magnetite and silica contents) and rheological characteristics (e.g., bedding attributes and competency contrasts between individual beds) in the sedimentary horizons, and the relative proportions of which are variable. Some sulphide layers were mapped as the lateral termination of subconcordant quartz-sulphide veinlets (Figure 7.11). High-grade gold mineralization (>100 g/t Au) is restricted to the narrow quartz-sulphide veinlets and sulphide layers.



(a) Typical quartz-sulphide gold-bearing vein hosted within a mafic sill, showing sharp contacts, high sulphide content and quartz crystals oriented perpendicular to the margin . Rock slab from zone 3W, level 235. (b) Concordant, parallel layers of massive sulphides in a sedimentary horizon. Zone 20W, level 355. (c) Veining (dark material) and intense sericitic alteration within a QFP dyke, (d) Dismembered quartz-sulphide-gold-bearing vein (dark material) hosted within a foliated QFP dyke, yielding apparent shear-related vein relationships. Zone 3S, level 235.

Figure 7.9 – Photographs showing gold-related features



(a) Branching of veins defining the junction between the E-W-striking, steeply south-dipping A vein and the N-S-striking, shallowly east-dipping 10 vein. Vertical view of the pillar separating the A and the 10 stopes, level 145. (b) Lateral vein termination showing arborescent multi-branching veinlets. Vertical view of the northern termination of the N-S-striking, shallowly east-dipping 1 vein, level 235.

Figure 7.10 – Sketches of vein geometry from photographs

7.3.4.3 SE-NW veinlet arrays

Arrays of gold-bearing veinlets occur in close spatial association with the gold-bearing veins hosted by mafic sills. These 1-cm to 4-cm-wide quartz-sulphide veinlets strike consistently SE-NW and dip steeply to the NE, parallel to the QFP dykes (Figure 7.11). Individual veinlets are planar and exhibit a regular distribution, with spacings of 1 to 2 m. The SE-NW veinlets generally have a consistent width, but some are boudinaged in vertical section. The grade of the veinlets ranges from 1 to 10 g/t Au. Figure 7.11 shows the lateral variation of the mineralization styles, structural elements, gold-bearing SE-NW veinlet array, and equal area projections of structural data (lower hemisphere). The Figure 7.11 inset shows schematic progressive mineralization style transition from east to west and change of mineralization occurrences from quartz-sulphide veinlets to sulphide layers within the stratabound mineralization at the western end of the zone.

7.3.4.4 Veining within the QFP dykes

This style of mineralization occurs solely within the QFP felsic dyke group, where it forms multiple, millimetre-wide quartz and sulphide veinlets and disseminated auriferous pyrite Crosscutting relationships indicate episodic veining injections within the dykes.

Mineralization within the QFP dykes is a mine-scale phenomenon, but it is best developed in close spatial association with the gold-bearing veins. Systematic assays have revealed erratic gold grades ranging from nil to hundreds of g/t Au.

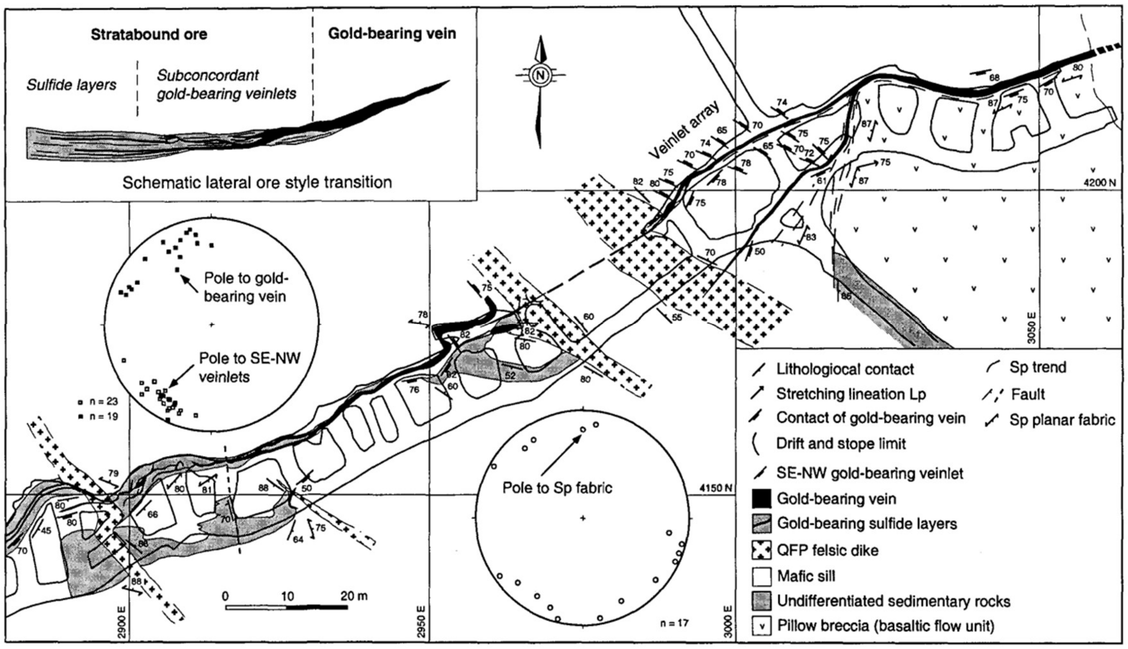


Figure 7.11 – Detailed geological map of the 20E-20W zone at level 355.

7.3.5 Vein and veinlet material

Veins and veinlets forming the different styles of gold mineralization share the same mineralogy, and there is no evidence for deposit-scale mineralogical zoning. Gangue minerals are dominated by gray quartz with minor amounts (< 2%) of calcite, chlorite and sericite. In some parts of the gold-bearing veins, milky quartz is abundant and forms subparallel bands ranging in thickness from 1 to 5 cm. These bands are superimposed on the massive gray quartz and are interpreted as evidence of multi-stage vein infilling. The amount of sulphide ranges from 5% (disseminated) to 80% (massive), with an average of 25%. The sulphide assemblage is composed mainly of pyrite and pyrrhotite, with lesser amounts of chalcopyrite and sphalerite and traces of galena and arsenopyrite. In addition to the base metals Cu, Zn,, Pb, veins contain lesser amounts of As, Ag, Au and Hg, with an Au/Ag ratio ~ 0.5 (Table 7.1).

Table 7.1 – Metallic composition of mined and crushed gold-bearing rock

Metal	Composition	Metal	Composition
Cu	0.36%	Au	9.7 g/t
Zn	0.13%	Ag	21.0 g/t
As	60 g/t	Hg	0.24 g/t

Values determined by the Centre de Recherches Minérales (CRM) from a composite sample of vein and stratabound mineralization styles.

Gold, as observed in polished thin section, occurs mainly as micron-scale inclusions in sulphides (pyrite, chalcopyrite, sphalerite and pyrrhotite) and in smaller amounts as fine fracture fillings in pyrite.

7.3.6 Textures of the veins and veinlets

The gangue has a mainly massive texture due to interlocking of fine to coarse anhedral quartz grains. Comb-texture, defined by quartz crystals oriented perpendicular to the vein walls, is also common for the gold-bearing veins irrespective of their attitude (see Figure 7.9a), and is the dominant texture of the SE-NW veinlets. This texture indicates open space filling (e.g., Dowling and Morrison, 1989). The sulphide assemblage within the gold-bearing veins displays three main mesoscopic textures: (1) net texture, where sulphides surround millimetre-scale quartz crystals; (2) massive centimetre-scale lenses within the quartz matrix; and, (3) millimetre-thick bands of massive sulphides along the vein contacts.

7.3.7 Wallrock alteration

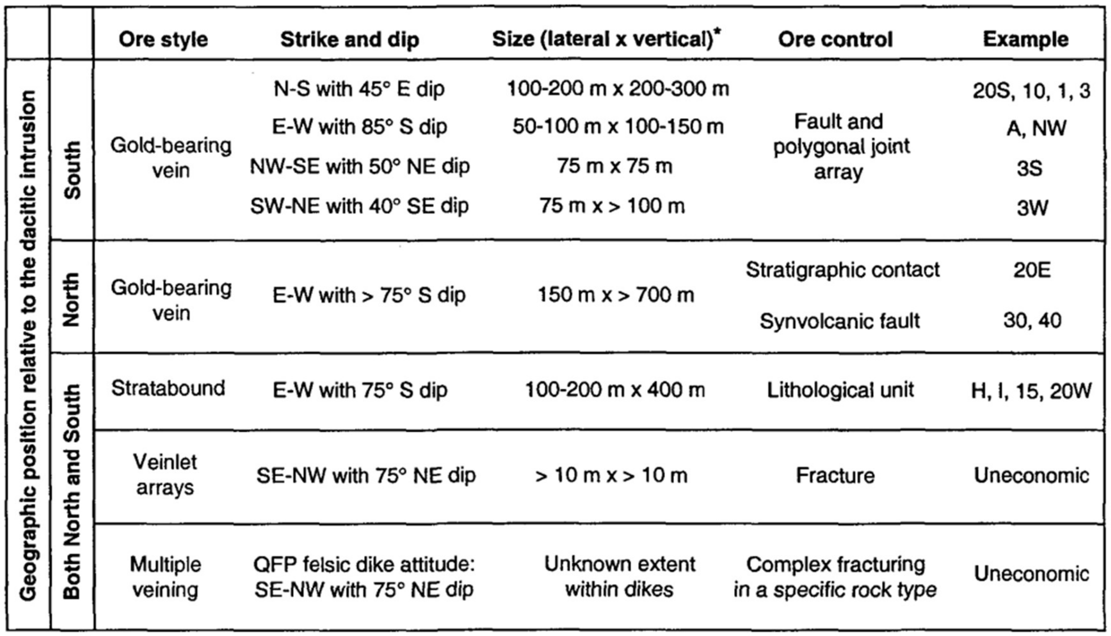
Most gold-bearing veins lack visible wall rock alteration. Where developed, alteration defines an irregular millimetre-scale envelope of sericite and/or chlorite at the contact between the vein and the host rock. Mineralized QFP felsic dykes are affected by intense and pervasive sericitic alteration. The QFP dykes are preferentially sericitized relative to the surrounding wall rock; this alteration phenomenon is developed at the deposit scale. The SE-NW veinlets have margins characterized by a 1-2 cm wide band of chloritic alteration in which a subvertical planar fabric (Sp) is generally well developed. No significant visible alteration is associated with the stratabound mineralization style.

7.3.8 Geometry of the deposit

Gold mineralization with economic potential is mainly restricted to the volcano-sedimentary succession bounding the dacitic intrusion (see Figure 7.7a and Figure 7.7c). At the mine-scale, mineralized zones are scattered within a 1-km^2 surface area.

Physical characteristics of the mineralized zones are summarized in Table 7.2. For the gold-bearing veins, there is a substantial difference in attitude and size depending on their position relative to the dacitic intrusion. North of the dacitic intrusion, the veins strike mainly E-W with a steep dip (> 75°) to the south. Their lateral extent ranges from 100 to 200 m with a vertical extent in excess of 700 m, yielding a ratio (lateral/vertical) of about 1:7 (see Figure 7.7a and Figure 7.7b).

Table 7.2 – Characteristics of different mineralization styles relative to the dacitic intrusion.



		Ore style	Strike and dip	Size (lateral x vertical)*	Ore control	Example
Geographic position relative to the dacitic intrusion	South	Gold-bearing vein	N-S with 45° E dip	100-200 m x 200-300 m	Fault and polygonal joint array	20S, 10, 1, 3
			E-W with 85° S dip	50-100 m x 100-150 m		A, NW
			NW-SE with 50° NE dip	75 m x 75 m		3S
			SW-NE with 40° SE dip	75 m x > 100 m		3W
	North	Gold-bearing vein	E-W with > 75° S dip	150 m x > 700 m	Stratigraphic contact	20E
					Synvolcanic fault	30, 40
	Both North and South	Stratabound	E-W with 75° S dip	100-200 m x 400 m	Lithological unit	H, I, 15, 20W
		Veinlet arrays	SE-NW with 75° NE dip	> 10 m x > 10 m	Fracture	Uneconomic
		Multiple veining	QFP felsic dike attitude: SE-NW with 75° NE dip	Unknown extent within dikes	Complex fracturing in a specific rock type	Uneconomic

* Measured on longitudinal sections parallel to gold-bearing zones

7.3.9 Relationships between mineralization styles

Although there are different sets of gold-bearing veins, all the veins have the same mineralogical composition. Branching and merging between different sets of gold-bearing veins is also observed indicating that these veins, regardless of their attitude, result from the same mineralizing episode. As the different gold-bearing styles share the same gangue and sulphide content, and relative proportion, a common origin is proposed for all mineralization styles. Furthermore, observed transitions from gold-bearing veins to stratabound mineralization are also consistent with a single, albeit prolonged and episodic, mineralizing system. In the case of the H and NW zones, the lateral transition in mineralization style is sharp and coincides with the intrusive contact of the dacitic body (see Figure 7.7a). In the case of the 20W and 20E zones, the transition from a single gold-bearing vein to the stratabound gold-bearing unit is progressive (see Figure 7.8).

7.3.10 Structural geology

Deformation affecting the host rocks is expressed by (1) the subvertical attitude of the volcanic strata; (2) the development of ductile planar and linear elements; (3) local mesoscopic folds; and (4) subhorizontal extensional calcite veinlets. These features are related to the regional deformation event. Subsequent deformation includes local shear zones and late brittle faults.

7.3.10.1 Planar and linear elements

A planar fabric, referred to as Sp (for principal surface), affects all rock types, including the hornblende-rich dykes. The Sp fabric is commonly well-developed within the sedimentary horizons, where it is defined by the alignment of phyllosilicates (chlorite and sericite). In other rock types, its development is typically weak. The mafic sills, which host many gold-bearing veins, display generally well-preserved gabbroic textures and lack a planar fabric, implying that the primary features are well-preserved. The Sp fabric strikes predominantly E-W with a subvertical dip (see Figure 7.8b). Variations of the Sp orientation are common, as illustrated by the density contour (0-2%) along the equatorial plane of the stereoplot (see Figure 7.8b). This distribution does not represent a combination of different fabrics, as no crosscutting relationship was observed either underground or in thin section. Furthermore, the consistency of the geometric relationships throughout the mine is incompatible with polyphase deformation. Instead, progressive changes in orientation are observed and are related to competency contrasts induced by felsic dykes, and, in some cases, by gold-bearing lenses. Therefore, the internal deformation within the tilted sequence is weak and heterogeneous, i.e., mainly concentrated within the sedimentary horizons and in some places along lithological contacts.

A weak to moderately developed stretching lineation (Lp), defined by mineral elongation, is associated with the Sp planar fabric. Stretching lineation distribution shows a dominant vertical plunge (see Figure 7.8b) despite variations of the Sp direction. Subhorizontal and planar extensional veinlets (< 1 cm) of calcite, developed within the more strained rocks, are structurally compatible with the stretching lineations. All the characteristics of the planar and linear elements imply that the ductile deformation was dominated by a vertical elongation.

7.3.10.2 Folds

In vertical cross sections, the subvertical Sp planar fabric and the steeply south dipping sedimentary bedding (So) define a 10-30° angular relationship. This indicates that the host sequence is on the northern limb of a regional syncline, considering the southward-younging direction of the sequence (see Figure 7.8c). Subhorizontal E-W intersection lineations between the So and Sp surfaces indicate an E-W trending, shallow plunging fold axis (see Figure 7.8d). The amplitude of the fold is unknown. Mesoscopic folds are also developed locally within the sedimentary horizons. They exhibit irregular geometry from open to tight closure, with variable fold axis attitudes (see Figure 7.8e). As these folds occur in both foliated and unfoliated sedimentary horizons, they are best explained as slump structures, locally accentuated by ductile deformation.

7.3.10.3 Shear zones and late faults

Some NW-SE striking brittle-ductile shear zones (< 2 m thick) with a moderate to steep dip to the NE, crosscut all the rock types, the gold-bearing lenses and the structural elements related to the regional folding. The sense of movement is dominantly reverse, as established by S-C relationships (Berthé et al., 1979) and by centimetre-scale offsets of local markers. Late brittle faults, with apparent centimetre- to metre-scale offsets, were also mapped.

8. DEPOSIT TYPE

This section was adapted from Bonneville (2019).

Sleeping Giant is a sulphide-rich lode gold deposit of volcanogenic affinity. In a geochemistry study of the volcanogenic massive sulphide (VMS) deposits in the Abitibi belt, Gaboury and Pearson (2008) classified the Sleeping Giant Rhyolite as “F1 type”, which includes VMS deposits rich in gold and silver. Such deposits, which have the particularity of being spatially isolated, are probably the result of local hydrothermal processes (Gaboury, 2004). The Sleeping Giant deposit displays atypical characteristics of orogenic Archean deposits associated with major faults (Table 8.1). According to Gaboury (1999), gold-bearing veins formed in subhorizontal strata shortly after QFP injections but before the end of mafic magmatism. All rock types, including gold-bearing veins, have been affected by regional ductile deformation (folding of strata and development of schistosity) and metamorphism to greenschist facies.

Gaboury’s proposed volcanogenic model is based on the chronology of geological units, alteration types and mineralization, as well as the geometry of the deposit. Although imperfect, this model best explains the genesis of the different types of gold mineralization encountered in the Sleeping Giant mine.

Table 8.1 – Mineralization characteristics associated with the Sleeping Giant deposit compared to typical Archean deposits in the Abitibi (Gaboury, 1998)

Feature	Volcanic Affinity (Sleeping Giant)	Structural Affinity (Archean deposits)
Au/Ag ratio	Weak: < 1 (± 0.5)	1 - 10
Sulphides	Po, Py, Cp, Sp	Py, Po, Asp rich
Metals	Cu, Zn rich	Cu, Zn poor
Alteration	Sericite/chlorite	Carbonate, albite, chlorite
Matrix material	Low carbonate content	Often carbonate-rich
Control	Volcanogenic origin	Orogenic origin
Geometry	More complex near paleosurface	No significant variation
Relation to deformation	No structural relation to regional deformation	Structural relation to regional deformation

9. EXPLORATION

The issuer has not performed any relevant exploration work since it acquired the Project in 2016. Previous exploration programs are presented in Item 6 (History).

10. DRILLING

This section summarizes the drilling methodologies and procedures from past owners' programs and Abcourt's recent drilling programs, based on information available to the Author (Olivier Vadnais-Leblanc, QP).

Since acquiring the Property, Abcourt has completed three underground diamond drilling programs (in 2020, 2021 and 2022). Abcourt intends to conduct more drilling as part of its future exploration work on the Property.

Historical drilling amounts to 8,177 drill holes for 1,162,797.63 m with 287,185 samples, and Abcourt's drilling amounts to 94 drill holes for 9,281 m and 1,203 samples.

10.1 Drilling Programs by Previous Owners

Very little information is available about the drilling procedures used during historical programs. Drill hole logs are available in assessment work reports filed with the government. The work and accompanying reports predate the establishment of Quebec's professional order of geologists and the implementation of strict QA/QC procedures and NI 43-101.

Recovery data are not available for most of the historical drilling programs. The analytical methods and the detection limits have varied over time and by company.

Drill collar positions were recorded using local mine grid coordinates and UTM coordinates. Collar locations were surveyed by mine surveyors and/or mining technicians. In many cases, downhole surveys were limited to acid-test dip measurements but have also included Tropari measurements (azimuth and dip) and, more recently, Reflex or Flex-It surveys.

The Author appreciates that the work was done according to the prevailing industry standards. Drill hole orientations (azimuth and plunge) appear adequate given the attitudes of the mineralized zones.

The results of Abcourt's recent drilling programs confirm the nature of the mineralization, the magnitude of the gold values, the thickness of the zones and their 3D locations.

10.1.1 Matagami Lake Exploration

The first drilling activities on the Property occurred between 1976 and 1982 (total of 12,900 m). The results led to the discovery of Zone A.

10.1.2 Perron Gold Mines / Aurizon Mines

Following the Property's acquisition in 1983 by Perron Gold Mines (later known as Aurizon Mines), various drilling programs took place at surface and underground, leading to commercial production from the mine between 1988 and 1991.

10.1.3 Aurizon Mines / Cambior

In 1991, exploration drilling resumed on the Property when Cambior acquired the mine. Between 1991 and 1993, 13,354 m of diamond drilling led to the discovery of four new zones: 20, 30, 40 and J-D. The results achieved by these programs made it possible to restart underground mining operations.

Between 1993 and 2008, extensive diamond drilling led to the discovery of zones 2, 3, 4, 5, 6, 7, 8, 9, 16, 18 and 50.

10.1.4 Cadiscor / North American Palladium

Following IAMGOLD's acquisition of Cambior, IAMGOLD decided to sell its Sleeping Giant assets to Cadiscor in 2008. Cadiscor immediately started a 90-hole program (18,669 m) to outline new mineral resources.

In 2009, North American Palladium acquired all shares of Cadiscor and became the sole operator at Sleeping Giant. They followed with various drill campaigns that summed up to 35,735 m in 2009, 77,264 m in 2011, and 36,746 m in 2012.

10.1.5 Maudore / Aurbec Mines

On August 27, 2013, Aurbec started an underground drilling campaign to increase the resources in zones 20, 30, 8 and 3, and to deepen the extension of Zone 16 and 785N below level 975. The drilling campaign was abruptly cut short in June 2014 after 26,781 m due to Aurbec's bankruptcy.

10.2 Drilling Programs Completed by the Issuer (Abcourt)

In a press release on November 10, 2022, Abcourt disclosed previously unpublished results from diamond drilling programs completed between 2020 and 2022. The results in the press release are presented in Table 10.1.

The results are from 94 drill holes totalling 9,281 m, drilled underground from levels 235 and 295 between November 2020 and May 2022. Of the 94 holes, 82 intersected at least one mineralized interval grading more than 2.0 g/t Au. Of the 9,281 m drilled, 770 m were sampled for a total of 1,140 samples. Samples were composed of whole core (i.e., no witness core was left in the box). Sample lengths were between 0.5 m and 1.0 m.

All samples were analyzed at the internal laboratory of the Sleeping Giant mine and were subject to a re-assay and quality control program using a certified and independent laboratory during the autumn of 2022 to verify and validate the results obtained from the internal laboratory. The results of the independent re-assays confirmed the validity of the internal laboratory results.

Underground diamond drilling programs were completed using AQTK diameter core.

A surveyor determined underground collar locations. Drill hole collars were aligned using front-sights and back-sights set by a surveyor (Figure 10.3).

The downhole plunge and azimuth were surveyed using a Reflex or Flexit tool.

At the drill rig, drill helpers placed the core into core boxes and inserted labelled wooden blocks every 3 m (Figure 10.4).

Technicians transported the drill core to the core logging facility, where geologists logged (described) it.

All the data was recorded using GeoticLog software. Logging included all pertinent information regarding lithologies, mineralization, structures (including orientations) and alteration types.

Figure 10.1 and Figure 10.2 display the underground drilling.

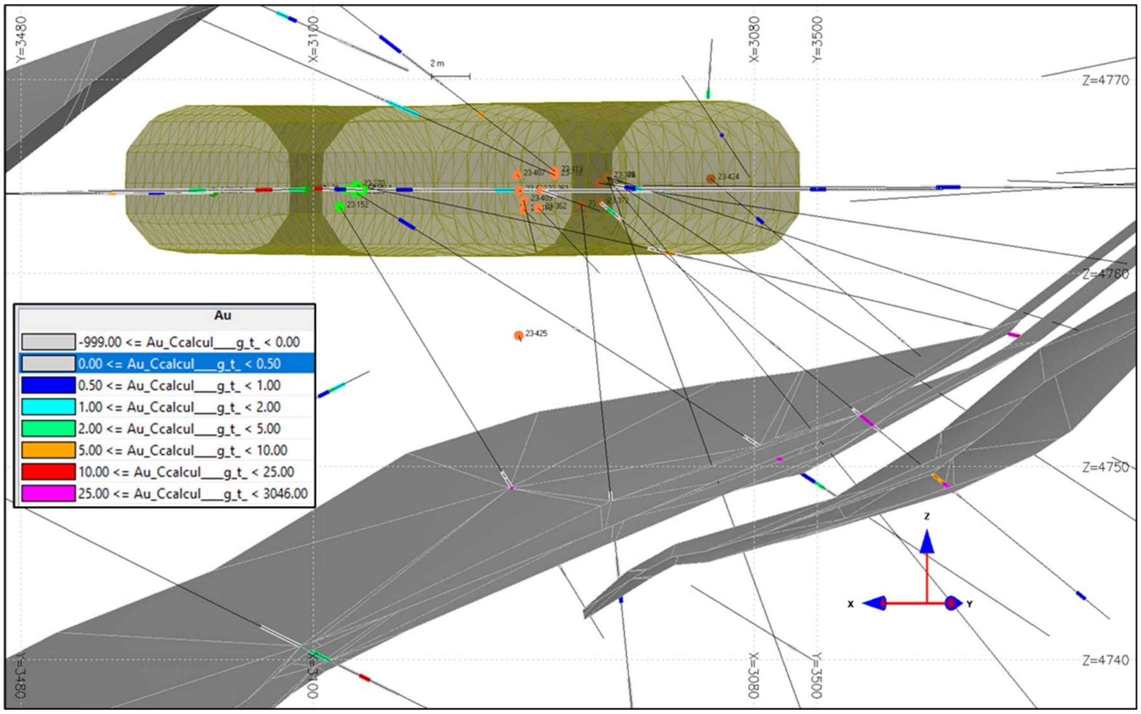


Figure 10.1 – Underground drilling (section view)

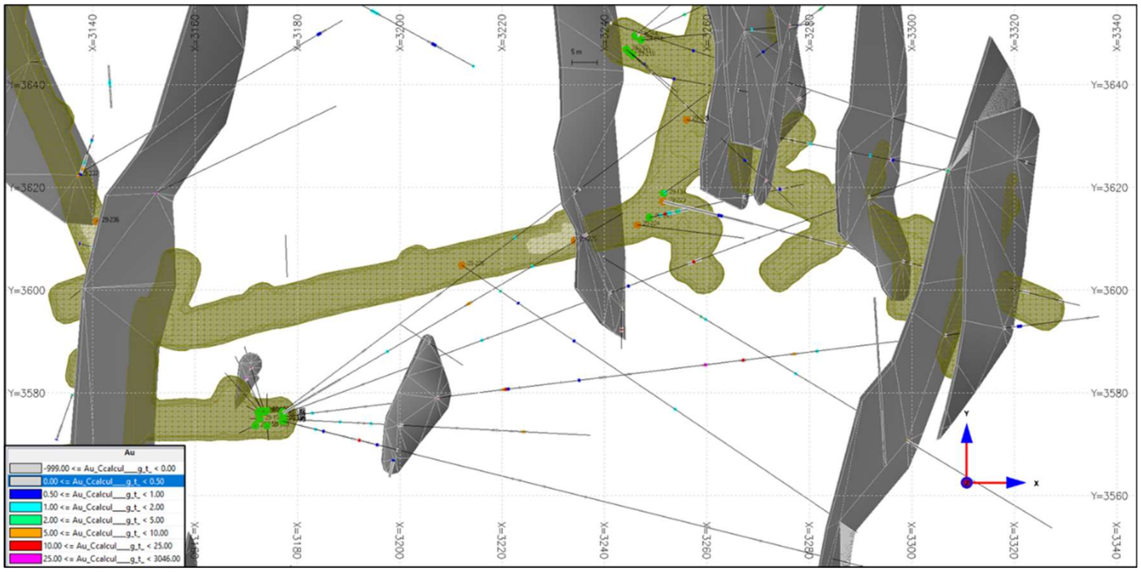


Figure 10.2 – Underground drilling (plan view)

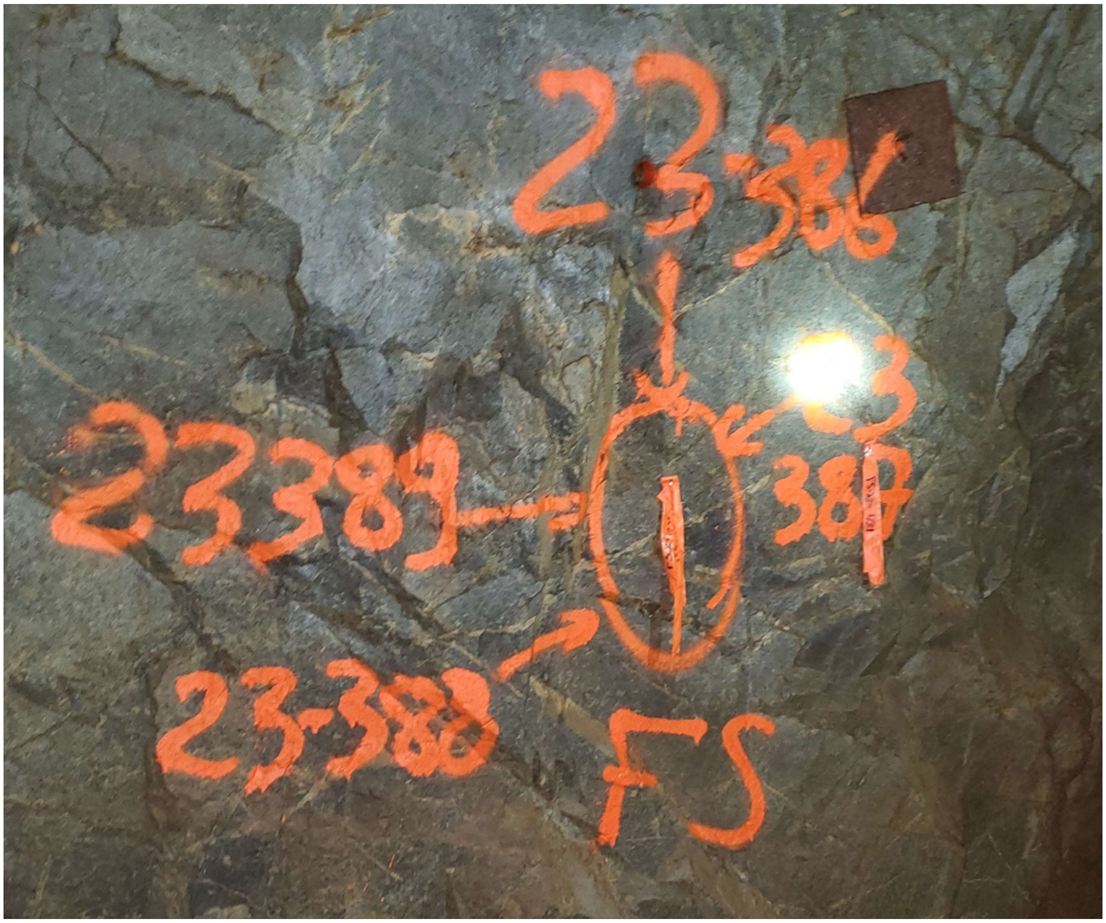


Figure 10.3 – Front-sight for underground drill hole collar surveying

Figure 10.4 – Wooden blocks placed every 3 metres by drill helper

Table 10.1 – Selected Significant Intervals from Abcourt’s 2020-2022 drilling campaigns

Drill Hole		From (m)	To (m)	Length (m)	Au (g/t)	Zone
23-342		0.00	0.50	0.50	2.73	Zone 3
23-343		11.30	11.80	0.50	4.82	Zone 4
23-344		29.40	30.00	0.60	7.45	Zone 4
23-347		No significant value				Zone 4
23-348		No significant value				Zone 3
23-349		6.10	6.80	0.70	7.98	Zone 3
23-350		0.90	1.40	0.50	4.12	Zone 3
	and	13.90	15.00	1.10	4.80	Zone 3
23-351		8.70	9.20	0.50	6.42	Zone 3
23-352		No significant value				Zone 3
23-353		No significant value				Zone 3
23-357		No significant value				Zone 3
23-358		12.60	13.40	0.80	4.33	Zone 3
	and	16.10	16.80	0.70	3.04	
23-361		26.60	28.00	1.40	14.15	Zone 3
	including	26.60	27.50	0.90	20.14	
23-362		20.00	20.70	0.70	8.34	Zone 3
23-363		34.00	34.50	0.50	120.68	Zone 3
23-368		No significant value				Zone 3
23-369		21.50	22.50	1.00	2.82	Zone 3
	and	23.60	24.40	0.80	125.52	Zone 3
23-370		16.90	20.60	3.70	7.19	Zone 3
	including	20.00	20.60	0.60	27.71	
	and	24.30	24.90	0.60	16.68	
23-371		1.40	2.20	0.80	2.09	Zone 3
	and	26.50	27.10	0.60	8.32	
	and	50.70	51.40	0.70	13.38	
23-372		0.50	1.00	0.50	2.14	Zone 3
	and	17.80	18.80	1.00	82.49	
	and	22.80	24.30	1.50	23.22	
	including	23.55	24.30	0.75	37.89	
23-373		17.00	18.10	1.10	5.28	Zone 3

Drill Hole		From (m)	To (m)	Length (m)	Au (g/t)	Zone
	and	27.70	28.50	0.80	4.17	
	and	66.00	66.70	0.70	2.62	
23-374		34.90	35.60	0.70	2.98	Zone 3
23-375		5.30	6.00	0.70	2.19	Zone 3
	and	34.20	35.20	1.00	18.21	
	and	40.15	41.20	1.05	2.33	
	and	57.00	57.50	0.50	3.86	
23-376		19.30	20.10	0.80	2.31	Zone 3
	and	35.70	36.30	0.60	2.03	
	and	68.00	68.55	0.55	3.15	
	and	73.80	74.30	0.50	12.10	
23-377		15.40	16.50	1.10	14.34	Zone 3
	including	15.40	16.00	0.60	19.04	
23-378		5.90	9.10	3.20	14.43	Zone 3
	including	8.10	9.10	1.00	32.64	
23-404		21.00	21.80	0.80	15.42	Zone 3
	and	22.80	23.60	0.80	3.37	
23-405		37.15	38.40	1.25	21.40	Zone 3
	including	37.15	37.95	0.80	25.69	
23-406		23.20	23.70	0.50	3.34	Zone 3
	and	28.50	29.00	0.50	16.37	
	and	43.90	45.00	1.10	13.15	
	including	43.90	44.30	0.40	32.65	
	and	51.11	51.61	0.50	2.56	
23-407		132.00	132.80	0.80	2.58	Zone 3
	and	150.20	150.70	0.50	4.16	
23-409		18.40	19.20	0.80	55.23	Zone 3
23-410		24.30	25.60	1.30	23.58	Zone 3
	including	24.30	25.10	0.80	35.34	
23-413		8.60	9.10	0.50	4.55	Zone 20
	and	18.00	18.50	0.50	4.69	
	and	21.90	22.40	0.50	14.83	
	and	27.00	28.70	1.70	3.43	
23-414		17.20	17.80	0.60	2.71	Zone Dac 5

Drill Hole		From (m)	To (m)	Length (m)	Au (g/t)	Zone
	and	36.70	37.20	0.50	2.35	
23-415		51.40	51.90	0.50	11.98	Zone Dac 5
	and	63.10	63.60	0.50	11.00	
	and	111.00	111.50	0.50	2.10	
23-418		100.10	100.60	0.50	4.76	Zone 3
	and	147.00	147.50	0.50	2.54	Zone 4
23-419		5.00	5.50	0.50	7.10	Zone 3
	and	53.20	53.70	0.50	2.78	
	and	55.50	56.00	0.50	2.45	
	and	118.70	119.20	0.50	7.91	
	and	124.20	125.00	0.80	2.98	
23-420		No significant Value				Zone 3
23-421		No significant Value				Zone 3
23-423		22.70	23.20	0.50	2.62	Zone 3
23-424		No significant Value				Zone 3
23-425		No significant Value				Zone 3
23-429		20.40	20.90	0.50	4.38	Zone Dac 5
	and	86.50	87.00	0.50	3.32	
	and	119.30	121.60	2.30	7.56	
	including	120.40	121.60	1.20	11.00	
	and	126.10	126.70	0.60	5.54	
	and	128.85	129.40	0.55	4.14	
	and	198.10	198.60	0.50	4.05	
23-432		46.25	46.85	0.60	8.28	Zone Dac 5
23-433		32.00	32.50	0.50	5.05	Zone Dac 5
	and	34.00	37.50	3.50	9.58	
	including	34.00	34.50	0.50	30.55	
23-434		16.40	16.90	0.50	3.35	Zone 20
	and	39.00	39.50	0.50	5.24	
23-435		29.20	29.60	0.40	7.76	Zone 20
	and	63.00	63.80	0.80	98.18	
	and	65.60	66.10	0.50	4.12	
	and	70.80	76.00	5.20	4.66	
	including	73.70	74.40	0.70	16.87	

Drill Hole		From (m)	To (m)	Length (m)	Au (g/t)	Zone
23-436		9.10	9.60	0.50	3.61	Zone 20
	and	73.00	73.85	0.85	18.02	
23-437		42.60	43.20	0.60	2.04	Zone 20
	and	54.10	54.80	0.70	6.56	
23-438		38.60	39.10	0.50	3.41	Zone 20
	and	53.20	54.20	1.00	2.20	
23-440		13.30	13.80	0.50	2.73	Zone 20
	and	47.20	47.70	0.50	2.60	Zone 30
	and	185.50	185.80	0.30	2.33	
23-441		52.90	53.40	0.50	2.16	Zone 20
	and	97.00	100.00	3.00	3.28	
23-442		20.40	20.90	0.50	7.54	Zone 20
	and	57.30	58.50	1.20	8.29	
	including	58.00	58.50	0.50	16.80	
23-443		27.55	28.05	0.50	12.52	Zone 20
23-444		18.00	18.60	0.60	50.19	Zone 20
	and	114.80	115.40	0.60	2.57	
	and	116.50	117.00	0.50	14.56	
	and	126.10	126.70	0.60	26.71	
23-445		10.80	11.30	0.50	4.65	Zone 20
	and	18.00	18.50	0.50	4.38	
	and	29.80	30.40	0.60	8.43	
	and	106.60	111.00	4.40	11.06	
	including	109.30	109.80	0.50	81.46	
23-446		25.10	25.60	0.50	2.95	Zone 20
	and	34.60	36.30	1.70	2.08	
	and	81.00	82.30	1.30	2.99	
23-447		119.90	121.10	1.20	2.31	Zone 20
23-448		2.30	2.60	0.30	3.47	Zone 20
	and	72.30	73.60	1.30	4.10	
	and	132.00	132.40	0.40	36.04	Zone 30
	and	135.00	135.40	0.40	3.50	
23-457		48.50	49.10	0.60	11.26	Zone 20
23-458		20.00	20.50	0.50	7.61	Zone 20

Drill Hole		From (m)	To (m)	Length (m)	Au (g/t)	Zone
	and	65.50	66.60	1.10	2.45	
	and	115.90	116.90	1.00	3.58	
23-459		24.15	25.60	1.45	2.76	Zone 20
	and	40.70	41.80	1.10	3.61	
	and	48.00	48.50	0.50	11.89	
	and	55.00	55.50	0.50	82.22	
23-460		20.70	21.10	0.40	7.27	Zone Dac 05
	and	114.00	114.40	0.40	5.50	
	and	140.00	140.50	0.50	2.47	
23-461		38.60	39.00	0.40	2.95	Zone Dac 05
	and	58.10	58.50	0.40	2.72	
	and	61.50	62.50	1.00	8.48	
	and	108.10	108.50	0.40	5.02	
	and	119.90	120.40	0.50	37.54	
	and	151.90	152.10	0.20	3.64	
23-462		136.50	136.90	0.40	2.29	Zone Dac 05
23-463		0.90	2.70	1.80	2.85	Zone Dac 05
	and	77.20	77.60	0.40	2.84	
	and	104.00	104.50	0.50	3.34	
	and	164.60	165.00	0.40	4.28	
23-465		No significant value				Zone Dac 05
23-466		109.50	111.60	2.10	3.77	Zone Dac 05
	and	140.70	141.20	0.50	5.90	
23-467		1.50	2.00	0.50	12.52	Zone Dac 05
	and	80.00	80.40	0.40	2.49	
	and	121.80	122.20	0.40	16.29	
	and	123.30	123.70	0.40	3.84	
	and	129.70	130.10	0.40	2.12	
23-468		3.30	4.35	1.05	6.80	Zone Dac 05
	and	23.80	24.20	0.40	4.40	
	and	89.40	89.90	0.50	20.26	
	and	106.40	106.80	0.40	2.47	
	and	115.70	116.30	0.60	5.50	
23-469		2.20	2.60	0.40	2.05	Zone Dac 05

Drill Hole		From (m)	To (m)	Length (m)	Au (g/t)	Zone
	and	8.80	9.20	0.40	2.34	
	and	33.50	33.90	0.40	8.77	
23-470		100.50	100.90	0.40	3.95	Zone Dac 05
	and	160.50	160.90	0.40	30.77	
23-471		99.60	100.00	0.40	3.71	Zone Dac 05
	and	129.70	130.10	0.40	10.28	
	and	180.20	180.60	0.40	3.73	
29-222		18.30	18.70	0.40	2.08	Zone 4
	and	39.60	40.40	0.80	13.38	Zone 5
29-223		47.50	48.20	0.70	77.38	Zone 5
	and	53.25	53.75	0.50	4.39	
	and	60.70	61.20	0.50	2.41	
29-225		5.80	6.05	0.25	4.45	Zone 4
	and	28.70	29.00	0.30	3.25	
	and	70.00	70.50	0.50	7.90	Zone 5
	and	75.40	75.90	0.50	8.55	
29-226		8.90	9.35	0.45	4.88	Zone 4
	and	88.65	89.15	0.50	2.36	Zone 5
29-231		118.60	119.10	0.50	2.81	Zone Dac 5
	and	165.75	166.25	0.50	91.77	
	and	168.80	169.30	0.50	4.64	
	and	173.50	174.00	0.50	9.93	
29-232		26.60	27.00	0.40	10.89	Zone Dac 5
29-235		9.20	9.70	0.50	52.83	Zone 4
	and	16.70	17.20	0.50	2.42	
	and	33.70	34.20	0.50	16.83	Zone 5
	and	54.20	54.70	0.50	2.70	
	and	57.60	58.10	0.50	9.46	
29-236		12.60	13.10	0.50	32.99	Zone 3
29-237		0.00	0.50	0.50	5.67	Zone 3
29-238		0.00	1.40	1.40	10.27	Zone 3
	including	0.50	0.70	0.20	43.95	
	and	5.50	6.10	0.60	2.97	
	and	119.00	119.70	0.70	8.44	

Drill Hole		From (m)	To (m)	Length (m)	Au (g/t)	Zone
29-239		21.00	21.50	0.50	5.31	Zone 3
29-243		66.00	66.60	0.60	11.80	Zone Dac 5
29-245		96.20	96.60	0.40	2.96	Zone 3
29-250		6.15	6.65	0.50	6.62	Zone 3
	and	43.20	43.70	0.50	6.49	
29-259		39.10	39.60	0.50	4.15	Zone 3
29-260		6.80	7.10	0.30	2.69	Zone 3
29-261		No significant value				
29-262		22.75	23.25	0.50	3.11	Zone Dac 5
	and	86.50	87.00	0.50	41.29	
	and	88.40	88.90	0.50	3.04	
29-263		3.50	5.00	1.50	4.04	Zone Dac 5
	and	5.90	7.65	1.75	33.11	
	including	5.90	6.80	0.90	54.89	
29-264		15.70	16.20	0.50	5.58	Zone Dac 5
29-265		21.79	22.39	0.60	4.38	Zone Dac 5
	and	23.44	23.93	0.49	6.41	

Notes:
1. The length represents the length measured along the drill core.
2. Assay results are not capped, but higher-grade sub-intervals are highlighted.

11. SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Historical Sample Preparation, Analyses and Security

Very little information is available before Abcourt's involvement in the Project. The QP believes that the sample preparation, analysis and security protocols for previous drill programs followed the generally accepted industry standards of the time. The overall historical results are of sufficient quality for exploration purposes. They have been confirmed to the same order of magnitude as the re-sampling programs completed in 2013 by InnovExplo (Verschelden and Jourdain, 2013) and the recent drilling, sampling and assaying completed by Abcourt.

11.2 Abcourt's Sample Preparation, Analyses and Security

The QP believes that the sample preparation, analysis, QA/QC and security protocols used by Abcourt followed accepted industry standards and therefore considers the analytical data valid.

Analyzed ore sections had to be more than 50 cm long, even if the mineralized zone was thinner than that. The maximum length of a sample was limited to 1 m. Core sampling was determined by the presence of a mineralized zone (usually within the extension of a known vein) that may contain gold. As part of its previous mining operation procedures, the standard practice at the internal laboratory was to analyze whole-core. Though unusual for exploration programs, the practice was justified in this case because mineralized zones defined by drilling were identified, mapped and analyzed daily by mine personnel over months or years.

The samples were individually bagged with the corresponding tag. The tag number was written on the bag before it was sealed. The bags were then placed in rice bags, and the bags were sealed. Abcourt personnel took the samples directly to the Sleeping Giant laboratory.

11.2.1 Sample preparation and assaying at the Sleeping Giant laboratory

Sample preparation was done onsite at the Sleeping Giant laboratory. The samples were crushed to 70% passing 10 mesh, then 250 g subsamples were pulverized to 90% passing 200 mesh (74µ). The reject portion, i.e., the unused fraction of the sample with particle size finer than -10 mesh or 1.7 mm (typically ~70%), was retained for future reference.

Gold analysis was carried out by fire assay on a 15 g pulp sample and finished with atomic absorption (code ALFA2). During the 2020-2022 drilling campaign no gravimetric finish was done on samples analysed at the internal laboratory. From 2023, samples returning a gold value greater than 10 g/t Au were re-assayed by fire assay and gravimetric finish.

11.2.2 Assaying at Techni-Lab (independent and certified laboratory)

From 2020 to 2022, pulps and rejects were assayed at Techni-Lab S.G.B. Abitibi Inc. ("Techni-Lab"), a division of ActLabs, in Sainte-Germaine-Boulé in Québec. It is an accredited external laboratory, ISO 9001 registered and ISO/IEC 17025 certified. Techni-

Lab used fire assay on a 30 g pulp sample and finished with atomic absorption (code ALFA2). Samples returning a gold value greater than 3 g/t Au were re-assayed by fire assay and gravimetric finish.

11.2.3 2022 Re-assaying program

All 2020 to 2022 drill hole samples were analyzed by the internal laboratory onsite at the Project. Unfortunately, no QA/QC samples were inserted during the procedure. Duplicate samples were sent to an external laboratory (Techni-Lab) for validation.

A total of 250 of the 1,140 samples, or 22%, were reanalyzed in the external laboratory. Of the 250 reanalyzed samples, 243 were from mineralized intervals grading more than 2.0 g/t Au and 167 of these, or 69%, were sent to the external laboratory as pulps and coarse rejects. The results of the independent re-assays confirm the validity of the internal laboratory results.

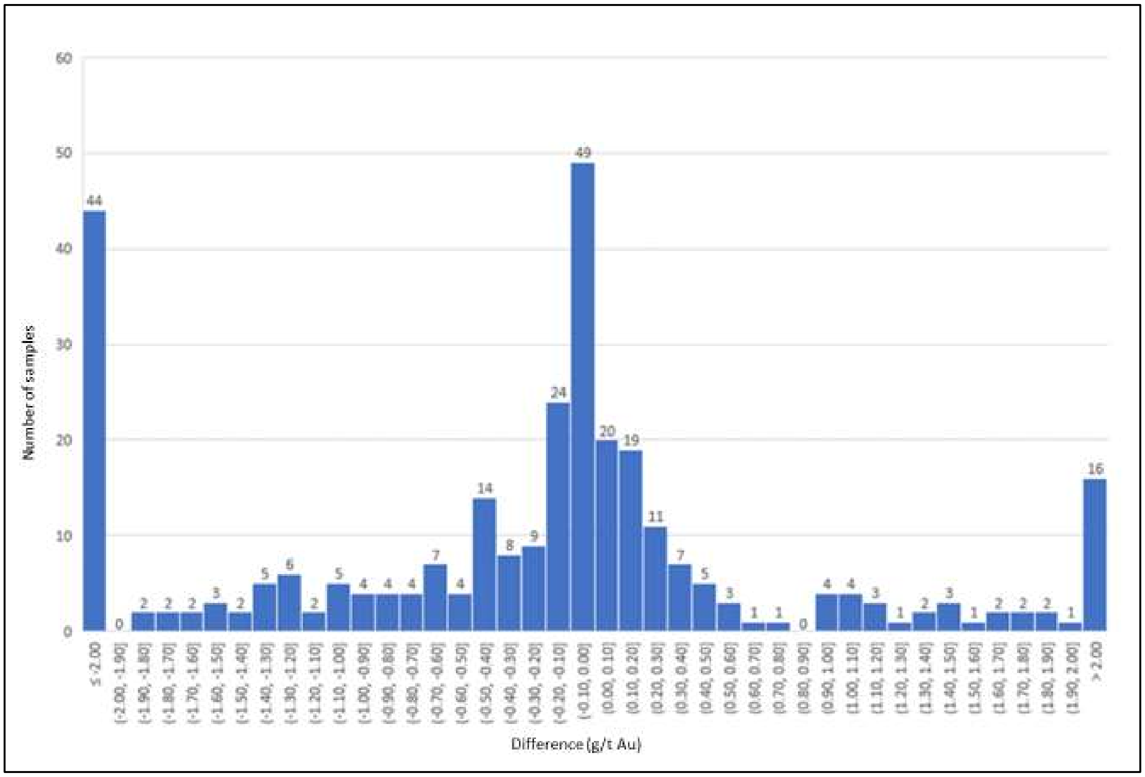
Duplicates made at the Techni-Lab facilities for validation purposes during Abcourt's internal analytical process were added to the new set of duplicates for a total of 308 duplicates.

The histogram in Figure 11.1 shows that the Abcourt internal laboratory often yields higher-grade results than the certified laboratory (200 samples (internal) vs 108 samples (Techni-Lab)); however, the grade difference is generally very low (Figure 11.2):

- 68 of the 308 pairs have a difference below 0.1 g/t Au
- 166 of the 308 pairs have a difference below 0.5 g/t Au
- Average difference of 0.67 g/t Au
- Median difference of 0.09 g/t Au

In general, the Q-Q plot correlation of the difference shows a 98% correlation (Figure 11.3).

The results of the independent re-assays confirm the validity of the internal laboratory results. Based on those results, the QP concludes that the samples analyzed by Abcourt at their internal laboratory are of sufficient quality for use in the 2022 MRE.



(negative values show higher results for the internal lab)

Figure 11.1 – Histogram of the differences in grade between original and re-assayed samples

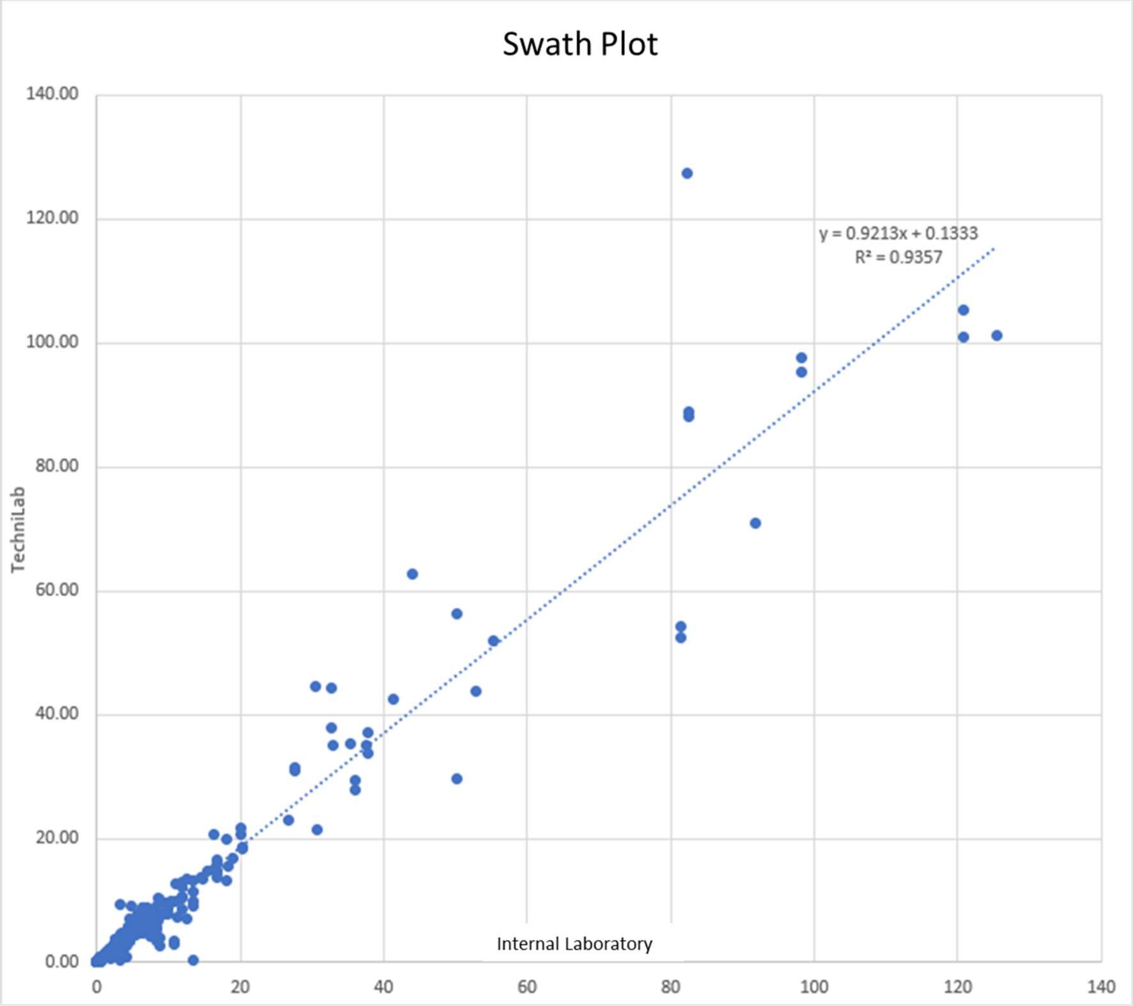


Figure 11.2 – Swath Plot of the grade differences (g/t Au): original vs re-assayed samples

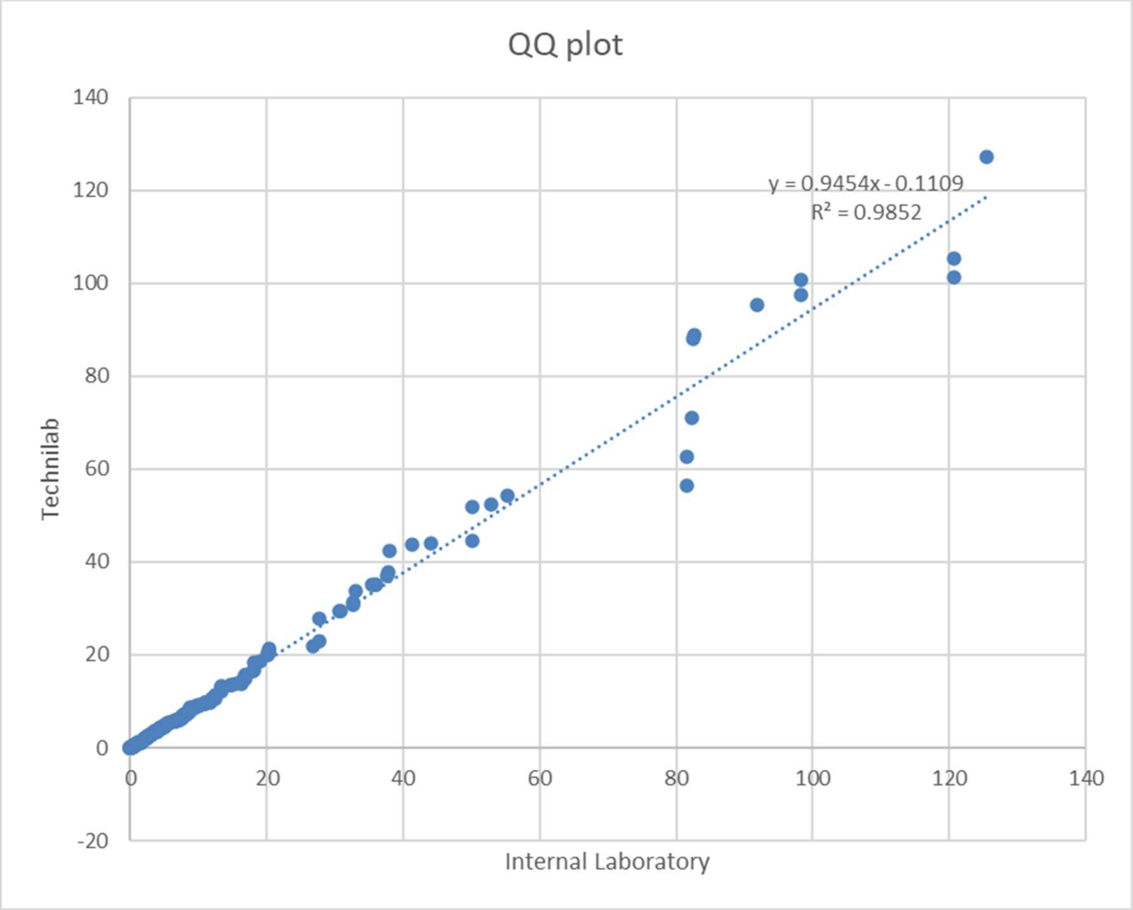


Figure 11.3 – QQ Plot of the differences (g/t Au): original vs re-assayed samples

12. DATA VERIFICATION

12.1 Site visit

InnovExplo's data verification included a field visit to the Property (mill, laboratory, logging and core storage facilities, offices) and an onsite review of the underground drill setup, underground developments and mineralized zones. Olivier Vadnais-Leblanc visited the Property on November 7 and 8, 2022, with chief geologist Mohamed Haithem Bennia. As whole-core samples (size AQTK) from the 2020 to 2022 drilling programs were sent to the laboratory, no remaining mineralized core (witness core) was available for review.

12.1.1 Laboratory

Keven Fortier, the technician in charge of the laboratory, led the tour of the Abcourt assaying laboratory. The visit included the crushers and grinder (Figure 12.1), the furnaces (Figure 12.2) and the atomic absorption instruments (Figure 12.3). The laboratory procedures were explained in full detail. The described procedures concerned the following: the preparation room, sample reception, crushers, grinder, riffle splitting, sample weighting, crucible preparation, fusion, pyro-analysis, gravimetric analysis, and cleaning.

Figure 12.1 – Crushers and grinders

Figure 12.2 – Furnaces

Figure 12.3 – Atomic absorption instruments

12.1.2 Pulp Storage

Pulps from the internal laboratory are stored in identified boxes stored on pallets in a container or in the core shack.

12.1.3 Mill

During the site visit, the mill was being cleaned. Abcourt planned to continue the cleaning until February 2023. After cleaning, the plant will be placed into care and maintenance mode until mining operations can feed the plant at a constant rate.

Figure 12.4 – Mill under maintenance (1)

Figure 12.5 – Mill under maintenance (2)

Figure 12.6 – Mill under maintenance (3)

12.1.4 Core shack

The core shack (Figure 12.8) is located just outside the main office building. It can accommodate three logging geologists at the same time.

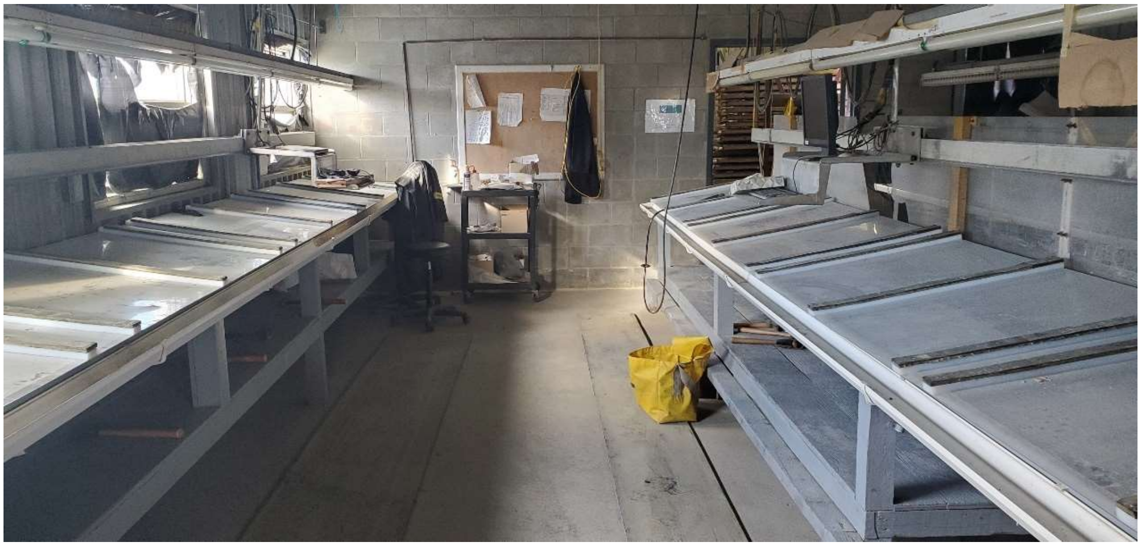


Figure 12.7 – Core shack (1)

Figure 12.8 – Core shack (2)

12.1.5 Core Storage

The remaining core is stored outside on metal or wooden core racks (Figure 12.9).

Figure 12.9 – Core racks

12.1.6 Underground

The QP visited the underground mine during the site visit. Drilling equipment has been left untouched since the mine closure (Figure 12.10)

Mineralized veins were observed underground (Figure 12.11, Figure 12.12 and Figure 12.13).

Figure 12.10 – Drilling Equipment

Figure 12.11 – Mineralized Zone 1

Figure 12.12 – Mineralized Zone 2

Figure 12.13 – Folded mineralized zone

12.2 Database

The database used for the 2022 MRE and the 2023 PEA contains a total of 8,433 surface and underground DDH, representing 1,185,868.63m of drilling and 288,388 assays.

The 2022 MRE and the 2023 PEA presented in the Report is supported by the following data:

- Historical diamond drill hole database – 8,339 holes (1,176,587.63 m);
- Abcourt 2020-2022 drilling results – 94 holes (9,281 m).

12.2.1 Historical diamond drill hole database

The 2022 MRE and 2023 PEA are largely supported by historical data. For this reason, much effort was made during the data verification process to obtain the highest degree of confidence in dataset quality and precision.

For the previous drilling program (2013-2014; 26,781 m of UG drilling), Aurbec was responsible for the QA/QC program. All samples were analyzed by the mine laboratory. Blanks and two (2) certified reference materials (CRM or "Standards") were inserted during the analytical procedure. Also, 588 pulps and rejects were sent to Agat Laboratory to duplicate the samples.

The QP has reviewed the methodology described in the 2019 technical report (Bonneville, 2019) and finds the conclusion appropriate. The QA/QC procedure satisfied the prevailing industry standards at the time. The QP has also reviewed the sample preparation, analyses and security item in the 2013 technical report (Verschelden, 2013) prepared for Aurbec. During this drilling campaign, a similar QA/QC procedure was used. Blanks and Standards were inserted by the mine geology department during the gold analysis procedure. Approximately 5% of the samples were sent as duplicates to ALS Chemex.

The Author has reviewed the methodology described in the 2013 technical report and finds the conclusion appropriate. The QA/QC procedure satisfied the prevailing industry standards at the time.

12.2.1.1 Abcourt 2020-2022 diamond drill hole database

Samples from the drilling campaigns of 2020, 2021 and 2022 were analyzed at the Sleeping Giant laboratory; however, QA/QC validation was not implemented during those analytical programs. To validate the 2020-2022 assays, Abcourt sent 22% of the samples, including 69% of the assays with a gold value over 2 g/t Au, for reanalysis at Techni-Lab, a division of ActLabs, which is an accredited external laboratory, ISO 9001 registered and ISO/IEC 17025 certified CCN (lab707), MELCC (lab375). Techni-Lab is located in Sainte-Germaine-Boulé (Quebec).

Independent results have been compared to the results from the Sleeping Giant laboratory. The correlation between results was determined to be adequate (Figure 12.14, Figure 12.15 and Figure 12.16), and assays from the Sleeping Giant laboratory were therefore used for the resource estimation.

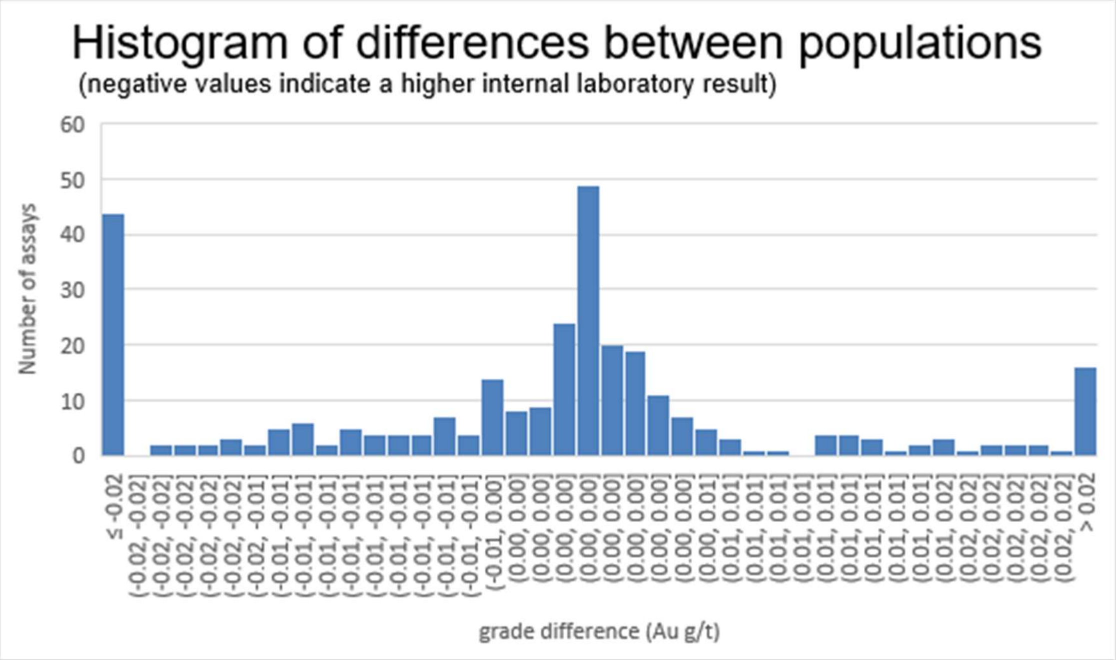


Figure 12.14 – Histogram of differences between samples from the internal laboratory and the independent laboratory

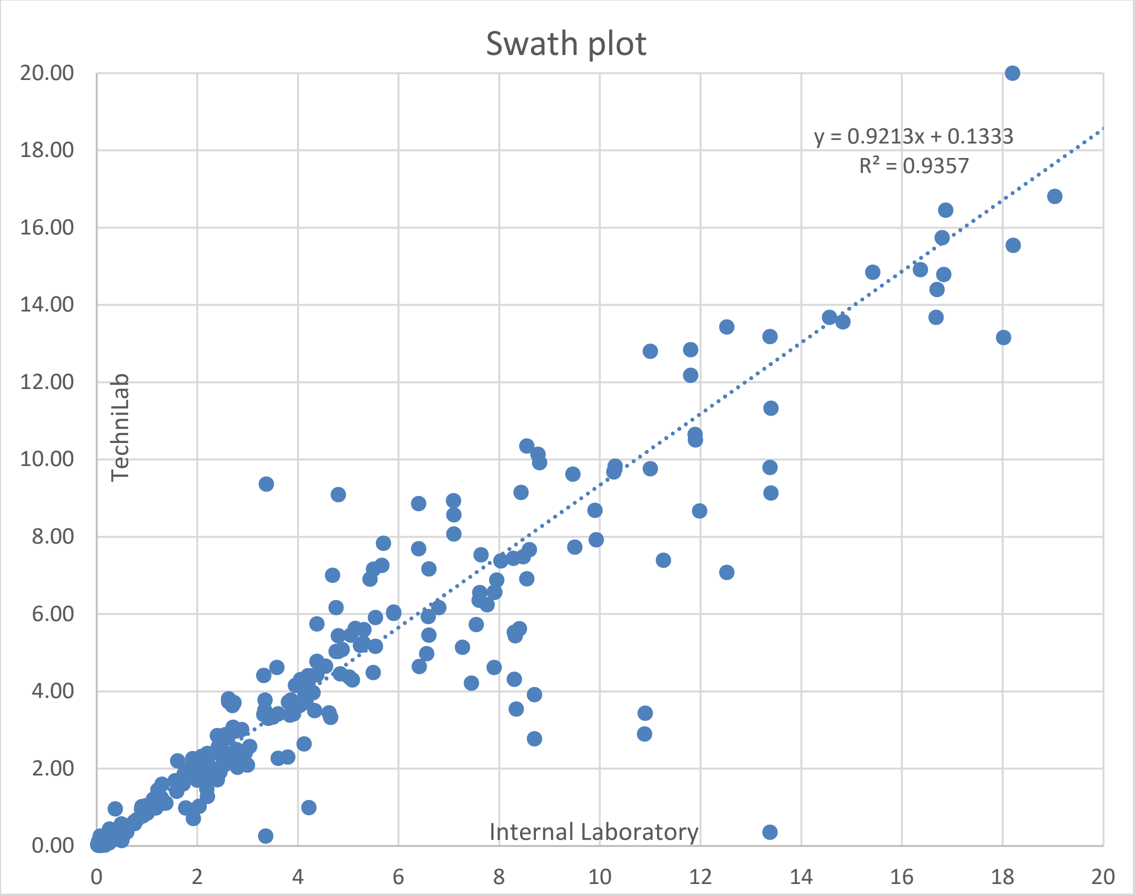


Figure 12.15 – Swath plot of assays (g/t Au) from the Sleeping Giant and Techni-Lab laboratories

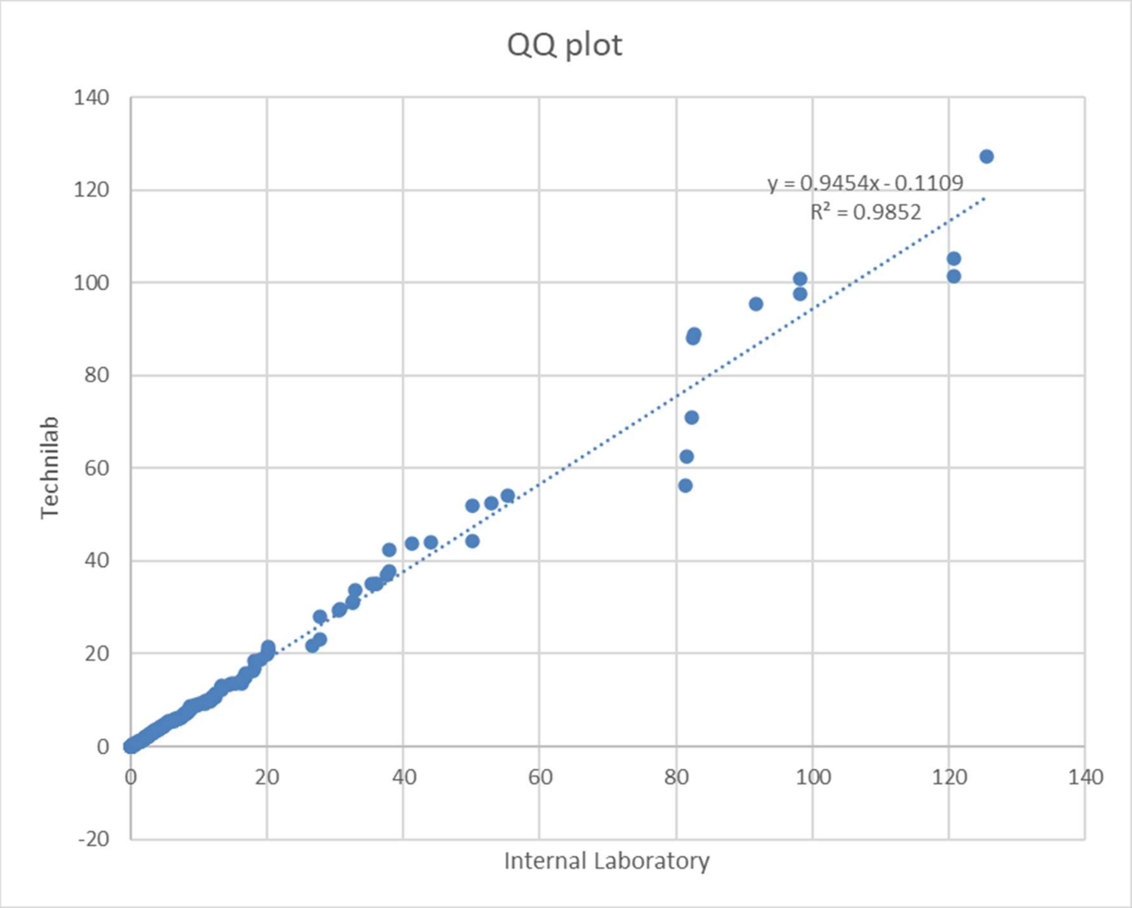


Figure 12.16 – Q-Q plot of assays (g/t Au) from the Sleeping Giant and Techni-Lab laboratories

12.3 Logging, Sampling and Assaying Procedures

Abcourt has three detailed procedures describing 1) the core logging steps; 2) how to enter the information in the database using GeoticLog; and, 3) the sampling procedure.

12.4 Conclusion

The QP considers the 2022 Abcourt drill hole database valid and of sufficient quality for the mineral resource estimate presented in Item 14 and for the 2023 PEA.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

Content in this section has been prepared by Soutex, with Guy Comeau of Soutex acting as the QP.

The Sleeping Giant concentrator was in operation from 1988 to 2014 and from 2016 until August 2022. During these periods it treated ore from the Sleeping Giant and other nearby mines.

The annual production data from 1993 to 2014, when processing Sleeping Giant ore, is shown in Table 13.1.

Table 13.1 – Annual Process Plant Production Data (1993 to 2014)

Year	Annual Tonnage	Hourly Tonnage	Daily Tonnage*	Head Grade (Au)	Solids Residues (Au)	Liquid Losses (Au)	Gold to Tailings	Gold Produced	Gold Recovery
	t/y	t/h	t/d	g/t	g/t	g/t	oz	oz	%
1993	90 630	32.2	773	8.50	0.54	0.090	1 840	22 896	92.56
1994	122 120	33.9	814	11.47	0.43	0.086	2 022	43 003	95.51
1995	124 522	32.5	780	12.45	0.37	0.081	1 817	48 042	96.35
1996	131 089	30.9	742	12.48	0.37	0.064	1 902	50 707	96.38
1997	147 940	32.4	778	10.59	0.28	0.071	1 762	48 628	96.50
1998	191 777	34.5	828	11.96	0.31	0.060	2 560	71 363	96.72
1999	208 407	33.8	811	11.96	0.29	0.064	2 560	75 482	96.71
2000	221 251	35.3	847	11.14	0.28	0.036	2 312	77 961	97.12
2001	214 068	34.3	823	9.60	0.28	0.038	2 190	63 859	96.68
2002	202 850	36.7	881	10.45	0.28	0.028	2 017	66 135	97.04
2003	176 493	37.9	910	12.09	0.32	0.039	2 000	66 609	97.09
2004	192 950	37.9	910	11.12	0.29	0.041	1 992	67 018	97.12
2005	148 623	37.7	905	10.63	0.29	0.050	1 583	49 210	96.89
2006	132 964	36.8	883	11.01	0.27	0.042	1 330	45 716	97.17
2007	170 391	36.6	878	12.52	0.27	0.055	1 784	66 826	97.40
2008	157 103	36.8	883	12.90	0.27	0.057	1 709	63 446	97.37
2009	32 822	36.9	886	5.85	0.2	0.037	250	5 920	95.95
2010	93 296	37.6	902	5.90	0.21	0.058	801	16 882	95.47
2011	74 153	34.8	835	6.31	0.19	0.028	520	14 528	96.54
2012	6 369	31.4	754	6.14	0.23	0.048	57	1 199	95.42
2013	4 920	27	648	5.44	0.13	0.022	25	835	97.11
2014	2 7521	25.5	612	6.32	0.17	0.036	182	5 407	96.74

*Daily tonnage is during operation. The plant was not in operation seven (7) days a week.

13.1 Metallurgical Testing

Historic and recent metallurgical test work data was not available for review and thus the QP is relying on available processing plant performance data presented herein to assess forecasted metallurgical performance assumptions. The QP notes the plant has been in operation for over 25 years and during this time has produced saleable gold ingots. Historical production can be considered a good predictor of projected future performance. The QP has not reviewed any metallurgical information that would suggest that future plant feed from the Sleeping Giant mine will be materially different from past plant feed.

13.2 Recovery Models

Mine geologists have indicated that the in-ground ore proposed to be mined is similar to the ore processed at Sleeping Giant in the past, and no changes in metallurgical plant performance are expected.

Annual production summaries for 1993 to 2014 were available for review, as were monthly production reports for 2001 to 2006 and 2010 to 2014.

The available yearly and monthly data were analyzed to determine if correlations could be identified between some variables (e.g., daily tonnages, solids residues, head grade, liquid losses). The main correlation identified was the one linking the Au solid residues to the head grade (Figure 13.1). Correlations calculated using annual data vs monthly data are similar. It is suggested to use the correlation determined from the monthly data set.

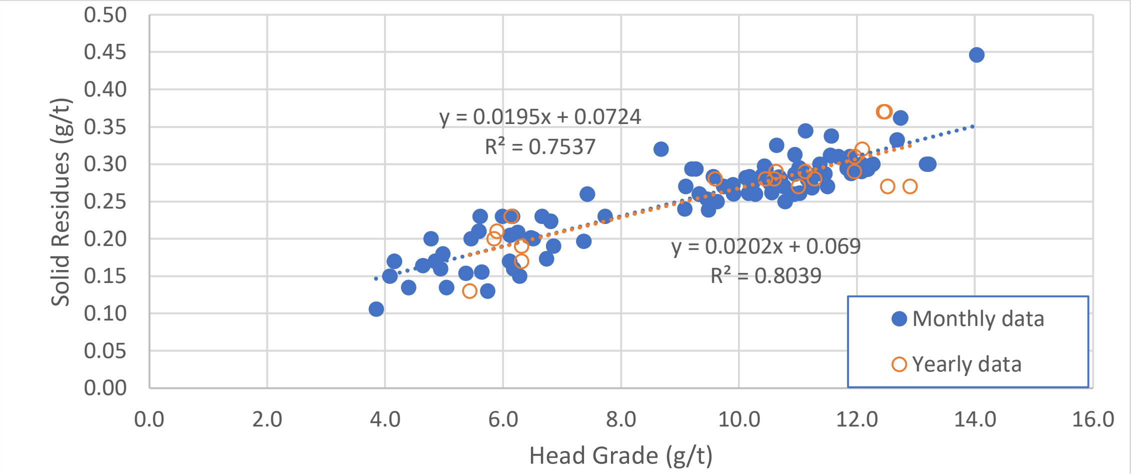


Figure 13.1 – Correlation between head grade (Au) and solid residues (Au)

No correlation was found to predict the liquid losses. It is suggested to use the historical data for a reference value. The average liquid losses calculated with the historic monthly production reports is 0.038 g/t Au.

The equation to predict the recovery based on historical production data is:

Recovery(%) = ((Head Grade – Solid Residues – Liquid Losses) / Head Grade)*100

Recovery(%) = ((Head Grade – (0.0202*Head Grade + 0.069) – 0.038) / Head Grade)*100

For example, if the Head Grade is 12 g/t Au, the recovery will be:

Recovery = ((12 – (0.0202*12 + 0,069) – 0,038) / 12) * 100

Recovery = 97.1%

14. MINERAL RESOURCE ESTIMATES

The Mineral Resource Estimate for the Sleeping Giant Property (the "2022 MRE") presented herein was prepared by Olivier Vadnais-Leblanc (P.Geo.) and Eric Lecomte (P.Eng.) of InnovExplo, using all available information. The main objective of the 2022 MRE mandate was to update the previous mineral resource estimate (the "2019 MRE"), which was published in a report titled "*Étude De Faisabilité Du Projet Géant Dormant, Rapport Technique NI 43-101*", dated July 31, 2019 (the "2019 FS") by Bonneville, (2019). The result of the 2019 FS was a mineral resource and mineral reserve estimate made from a polygonal mineral resource estimate. No three dimensional (3D) mineralized interpretation was used. The 2019 FS included measured, indicated, and inferred resources and proven and probable reserves for an underground volume.

The main objective of the 2023 PEA is to provide an early-stage assessment of the potential economic viability of the 2022 MRE.

The mineral resources presented in this item are not mineral reserves as they have not demonstrated economic viability.

The effective date of the 2022 MRE is December 12, 2022. This study does not include mineral reserves.

14.1 Methodology

The 2022 MRE was prepared using 3D block modelling and the inverse distance squared ("ID2") interpolation method for the Sleeping Giant deposit. Genesis software, version 2, release 21, was used to create the 3D mineralized vein shapes. Geovia Surpac 2022 was used to perform the interpolation and Deswik.SO was used to optimize the mineable stope shapes above the determined cut-off grade. Variographic studies were done using Isatis Neo Mining.

14.1.1 Drill hole database

All existing drill hole databases for the Property were compiled and merged for the 2022 MRE. The database used for the 2022 MRE contains 8,433 surface and underground diamond drill holes. Some exploration drill holes near the Sleeping Giant deposit were excluded from the 2022 MRE database. A local mine grid was used to locate the drill hole collar coordinates.

The database also includes conventional analytical gold assay results and coded lithologies.

The 8,433 holes cover the Property over an area of approximately 140 ha, within the limits of the 2022 MRE area (Figure 14.1 and Figure 14.2).

All header data (collar coordinates), down-hole survey data, lithological information and assay results were integrated into the Genesis database. Only the mineralized vein shapes and the composites were integrated into Surpac to estimate the resources.

The drill core intervals used for the interpretation contain 288,388 assays taken from the 8,433 drill holes, aggregating 1,185,868.63 m of core.

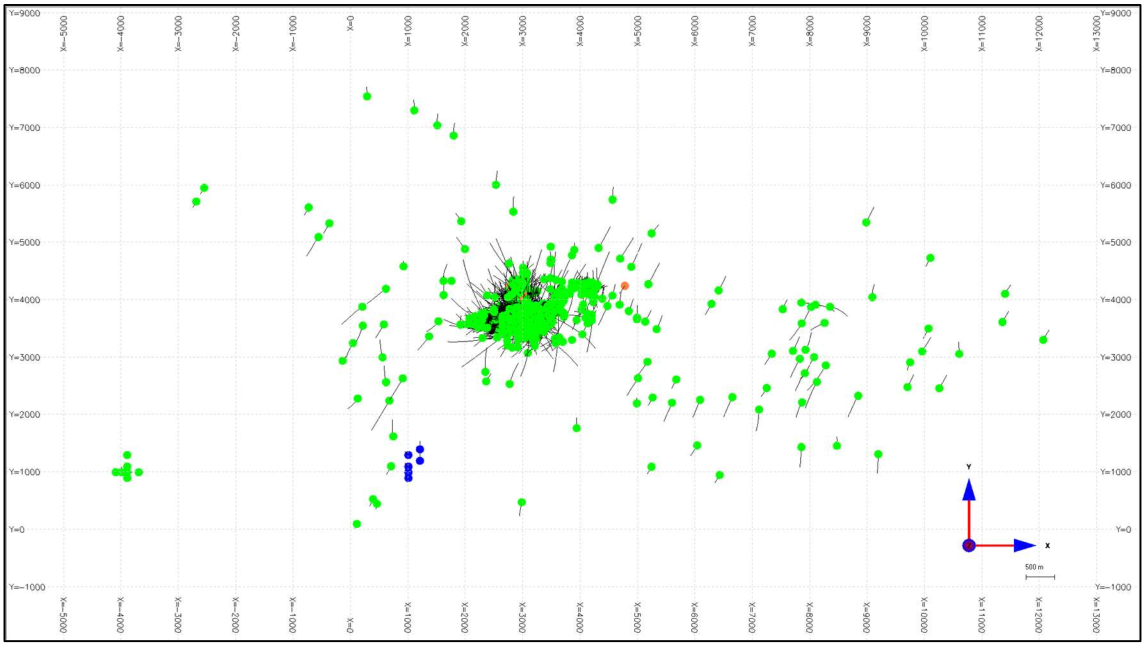


Figure 14.1 – Drill holes (plan view)

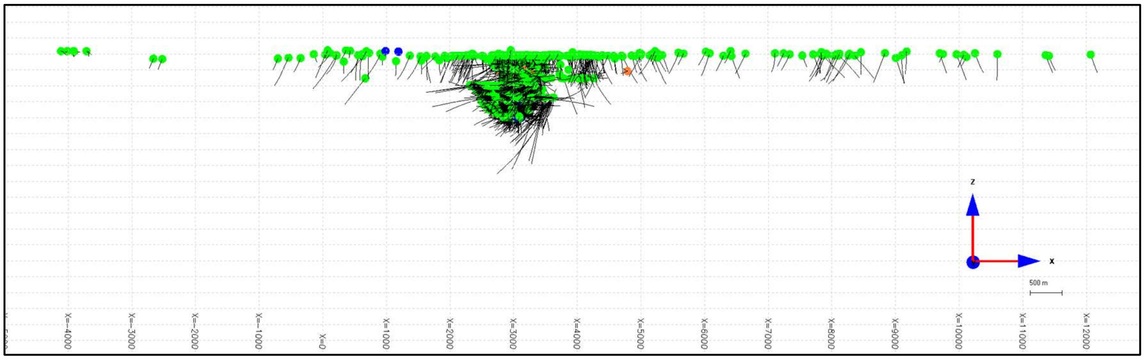


Figure 14.2 – Drill holes (section view)

14.1.2 Interpretation of mineralized zones

The mandate delivered to InnovExplo was to create a 3D interpretation of the Sleeping Giant deposit's mineralized system and update the resources using data from the holes drilled since 2019. The 3D model using a total of 846 vein wireframes created for this mandate is the first 3D interpretation made for the deposit (Figure 14.3).

Mineralized veins in the deposit are thin, with an interpreted average thickness of 0.7 m. The true in-situ thickness of the veins is often less than 0.7 m, but the assay length is rarely below 0.5 m. Veins are built with a minimum of one (1) assay, and the typical minimum assay length is 0.5 m. Out of 288,388 assays, 286,678 are over 0.5 m long, the majority being 0.5 m. The minimum modelling parameters used to construct the interpretation are 2.0 g/t Au over 0.5 m.

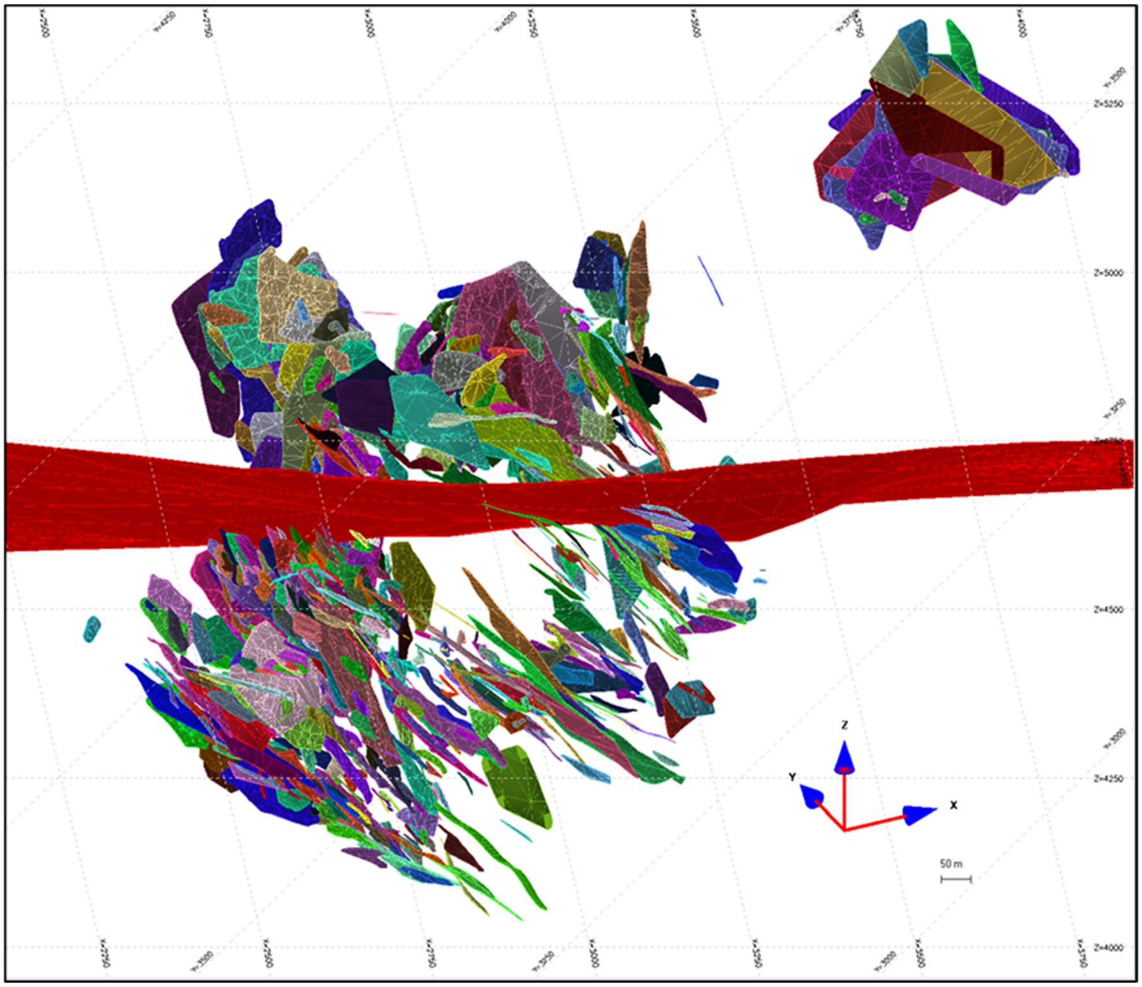


Figure 14.3 – 3D view of the veins model

To limit the wireframes, a 10-m margin has been set around the most external drill hole intercept. If a drill hole not selected for the interpreted vein is located in the margin area, the margin is automatically set at half the distance between drill holes. The 3D modelling was done using Genesis, V2.21.

14.1.3 Compositing

The gold assays were composited at 0.5 m within all defined mineralized zones to minimize any bias introduced by variable sample lengths.

Most sample intervals in the mineralized veins are 0.5 m long (Figure 14.4), and the average thickness of all veins is 0.7 m. Also, many veins are locally built on only one 0.5 m assay.

The total number of composites used in the drill hole dataset is 39,851. Composites have an average length of 0.48 m and the median length is 0.49 m. A total of 3,713 composites are based on assay intervals that are exactly 0.5 m. The smallest composites are 0.10 cm, and the longest are 0.67 m. All drill core sample composites less than 0.1 cm long were redistributed among the other composites of this interval (Figure 14.4). Compositing has been done in Genesis from drill hole intervals crossing veins.

Each mineralized zone solid (lens) was estimated separately using its own set of composites using hard boundaries. A grade of 0.0 g/t Au was assigned to missing sample intervals.

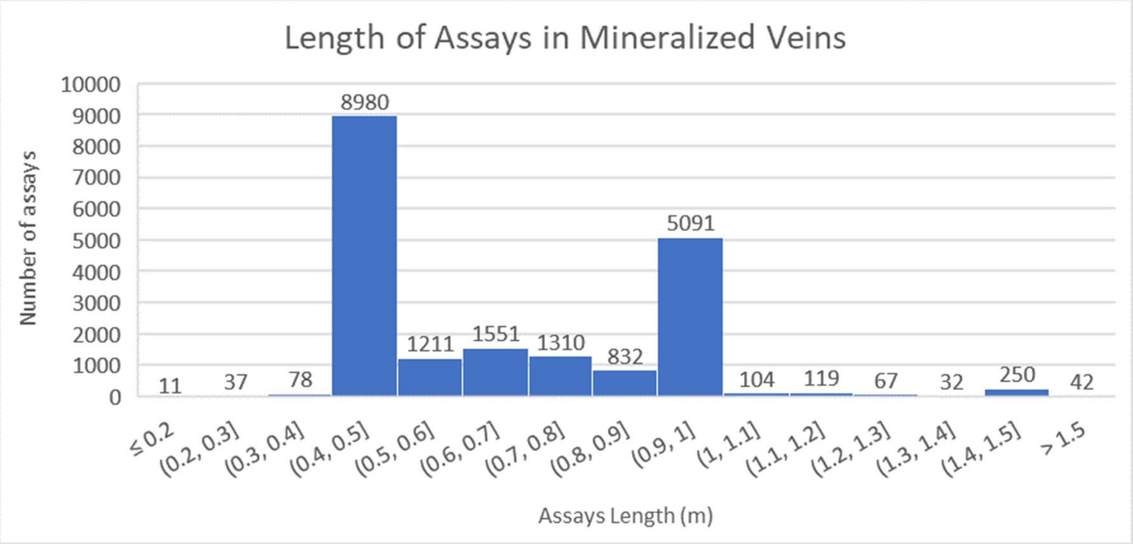


Figure 14.4 – Assayed core lengths within mineralized veins

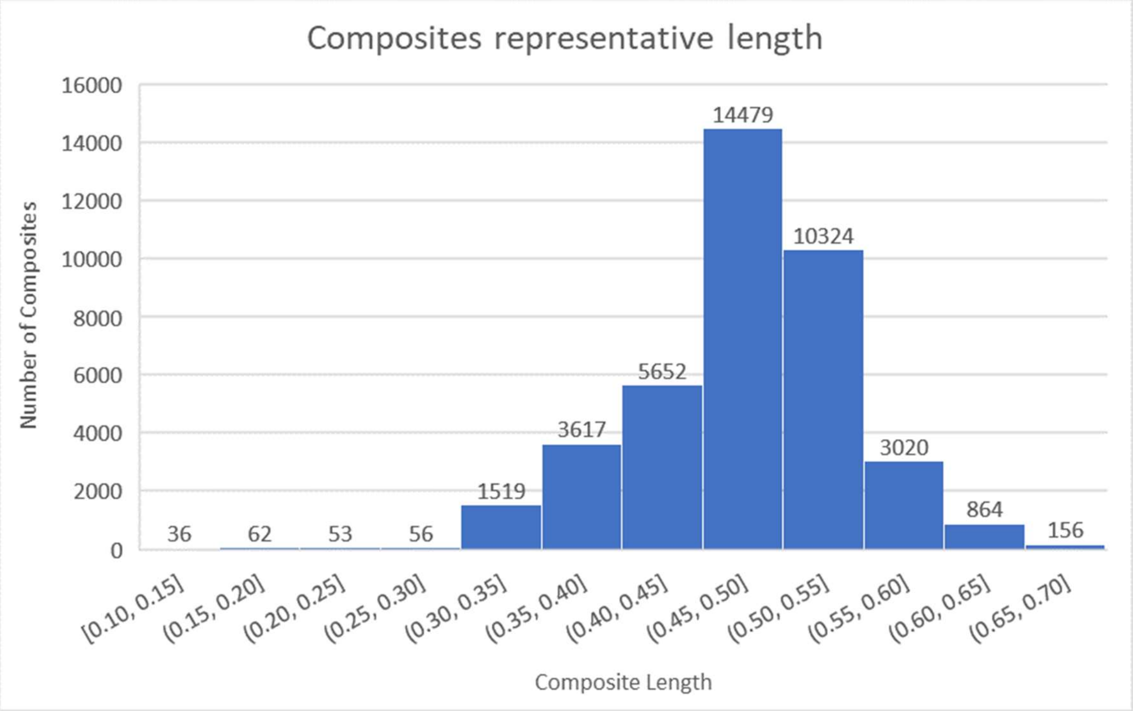


Figure 14.5 – Composite sample lengths

14.1.4 Capping

The deposit is divided by the "NW-SE fault". Capping grades have been established for veins occurring north of ("above") and south of ("below") the NW-SE fault (Figure 14.6 and Figure 14.7). The results are similar on both sides of the NW-SE fault, with a capping grade of around 95 g/t Au. For consistency, a single capping grade of 95 g/t Au has been used throughout the deposit (Figure 14.8 and Figure 14.9). With the capping set at 95 g/t Au, 484 composites (out of 39,851) were capped. This reduced the contribution of the 1% highest-grade composites from 18.6% to 11.5% of contained gold, a reduction of 7.1%. If the capping grade had been determined using the method of 10 times the average composite grade, it would be 90 g/t Au (Table 14.2).

Table 14.1 – Composites

General Composite Statistics			
	Uncapped composites	Composites Capped at 95 g/t Au	Difference
Average	9.0 g/t Au	8.2 g/t Au	-8.1%
Variance	555.5	272.0	-51.0%
Standard Deviation	23.6	16.5	-30.0%
Median	2.3 g/t Au	2.3 g/t Au	-
Max	1,351.4 g/t Au	95.0 g/t Au	-
Count	39,851	484 / 39,851	1.2%
Top 1% total Au contribution	18.6%	11.5%	-7.1%

Table 14.2 – Other capping information

Other info		
	Capping grade	Loss (%)
10x average grade method	89.6 g/t Au	-8.87
If 1 composite is capped	913.6 g/t Au	-0.12
If 2 composites are capped	868 g/t Au	-0.15

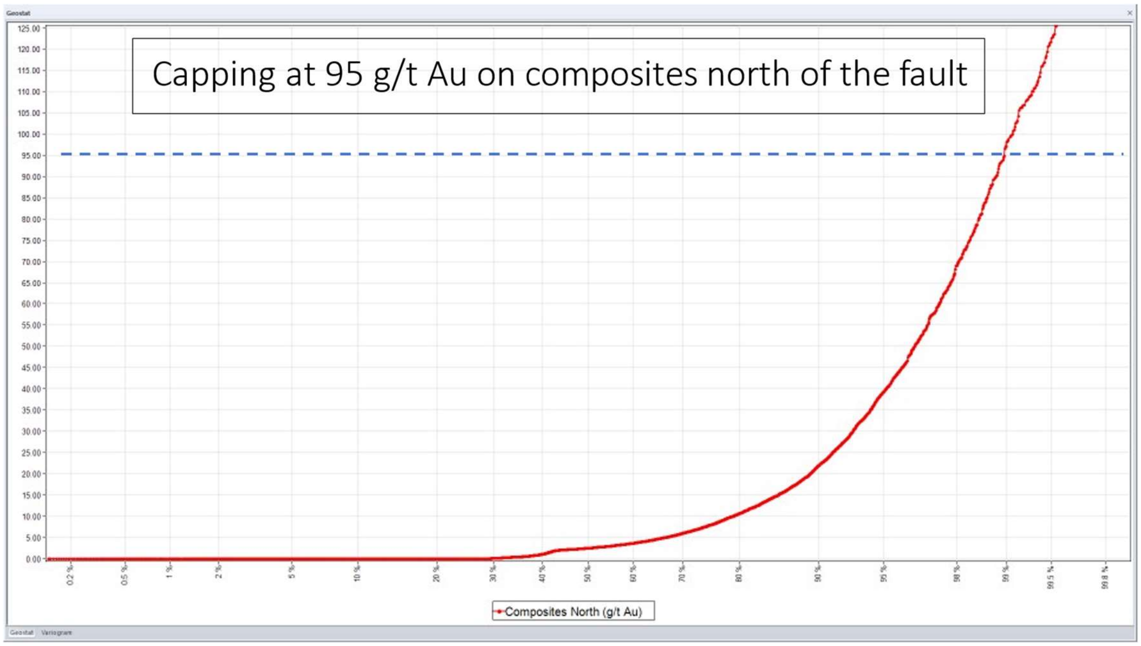


Figure 14.6 – Capping north of the NW-SE fault

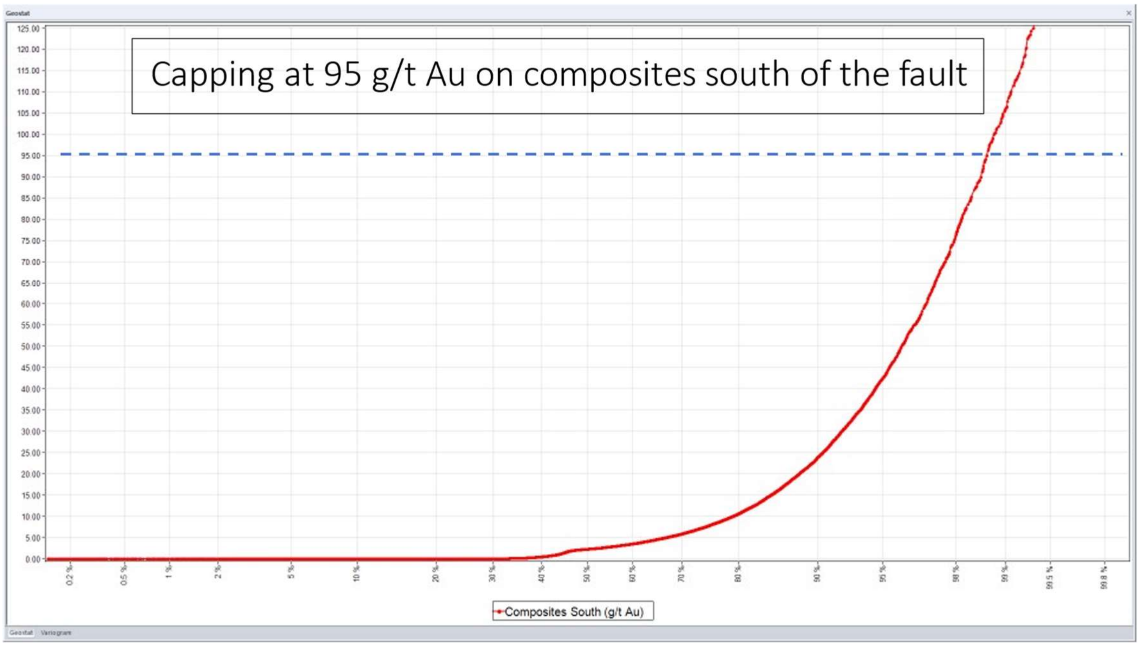


Figure 14.7 – Capping south of the NW-SE fault

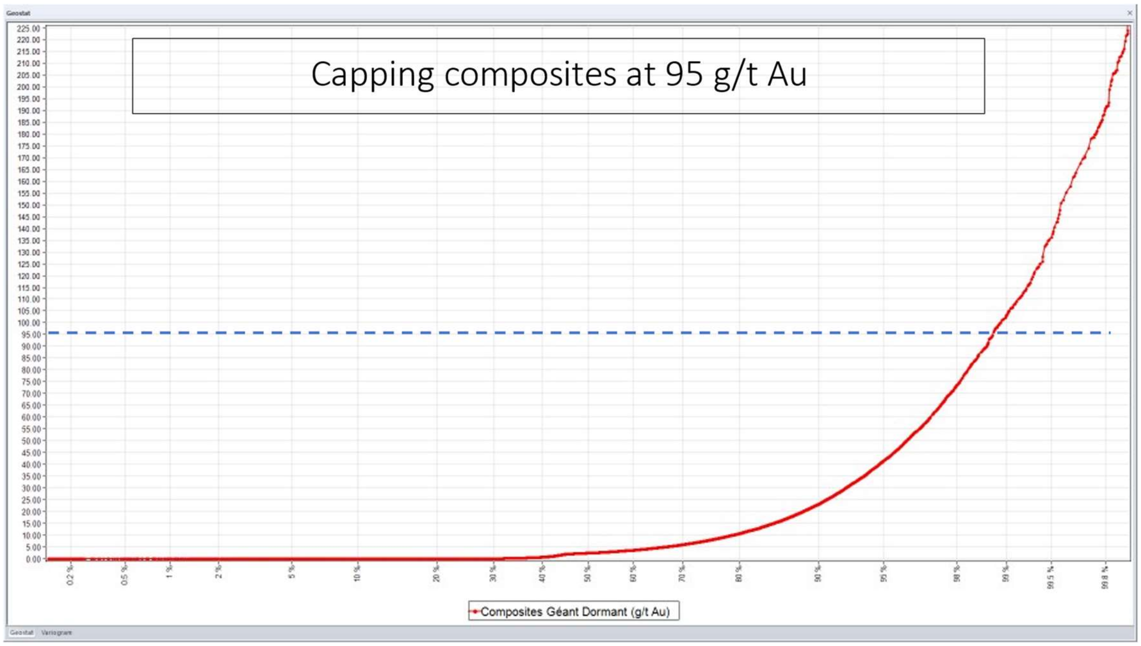


Figure 14.8 – Global capping at 95g/t Au

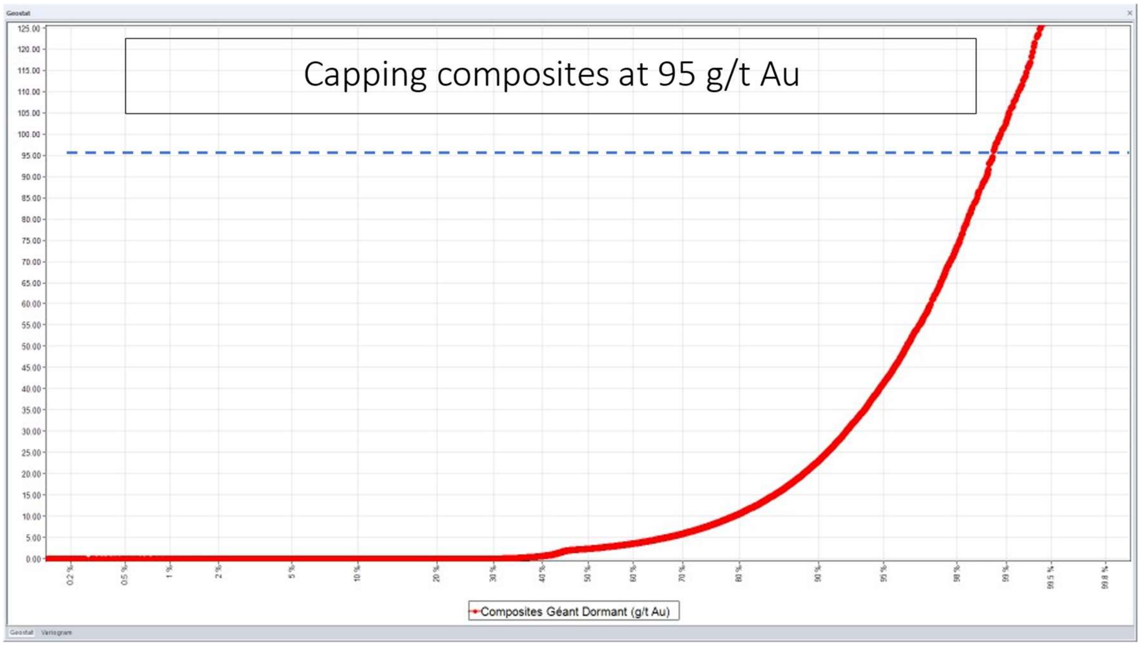


Figure 14.9 – Global capping at 95g/t Au (close up)

14.1.5 Variography

Variography has been evaluated independently for the parts of the deposit north and south of the NW-SE fault (Figure 14.10 and Figure 14.11).

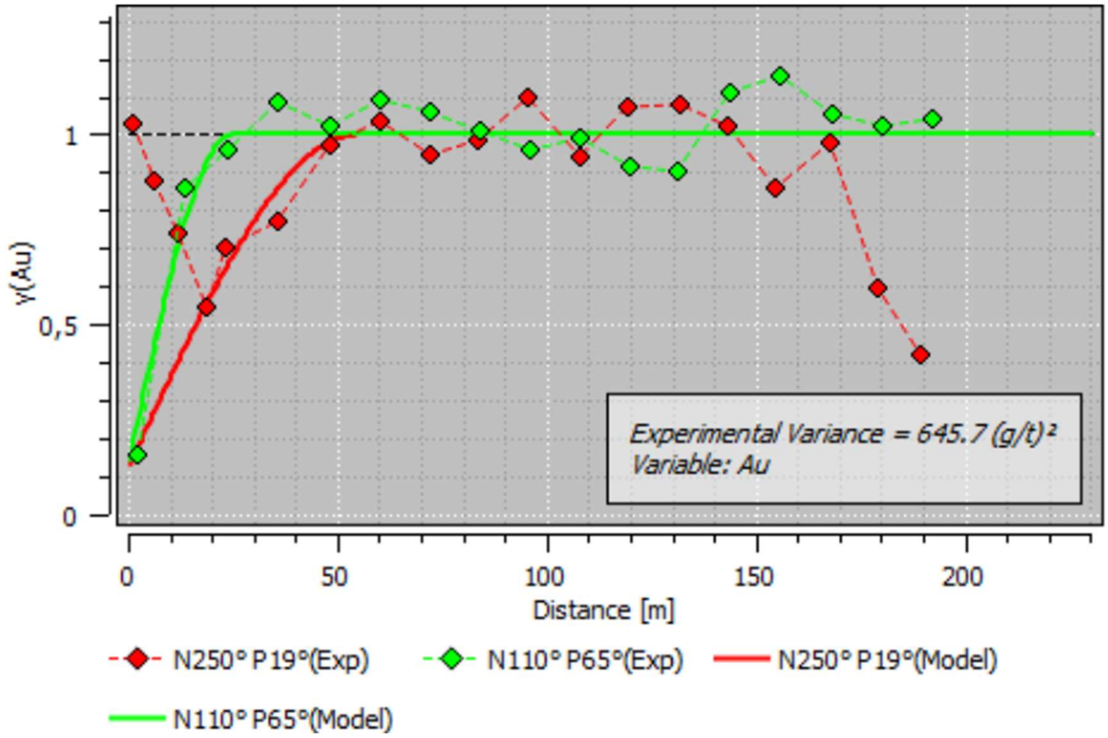


Figure 14.10 – Variography – north of the NW-SE fault

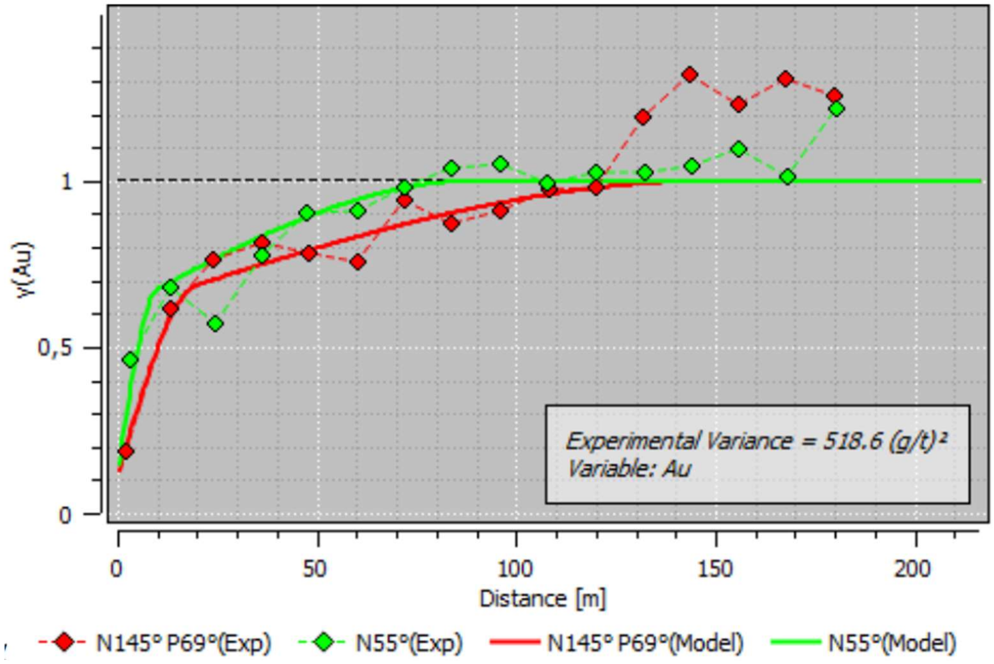


Figure 14.11 – Variography – south of the NW-SE fault

14.1.6 Bulk density

Cambior measured the density factor of the mineralized material in 2001 and 2002 by collecting and analyzing three samples per month from the material processed at the mill. The results varied between 2.8 g/cm^3 and 2.9 g/cm^3 for an average of 2.86 g/cm^3. A density factor of 2.85 g/cm^3 was therefore used by Cambior starting in December 2002 for both resource and reserve estimations (Asselin, 2008). The historical use of this density factor has allowed an acceptable reconciliation between the tonnage produced and the tonnage estimated by calculation (Jourdain et al., 2011). A global density factor of 2.85 g/cm^3 was used for the 2022 MRE.

14.1.7 Block model geometry

The deposit wireframes were used to constrain composite values chosen for interpolation and the mineral blocks reported in the 2022 MRE. A block model (Figure 14.12 and Figure 14.13) located on a local mine grid with block dimensions of 4 x 4 x 4 m in the X (east), Y (north) and Z (level) directions was placed over the wireframe models created for Sleeping Giant. Those blocks were locally sub-blocked down to 1m x 1m x 1m where needed. The block size was selected based on the geometry of the vein structures, the mining method (underground with the use of a stope optimizer), the borehole spacing, and the composite assay length. Veins have different dips and directions. The use of the stope optimizer helped determine a small sub-block size. Small blocks are necessary to better fill the interpreted mineralized veins with an average thickness of 0.7 m; otherwise, the voids created by bigger blocks would be classified as waste during the optimization process, generating dilution, which would in turn artificially diminish the number of mineable stopes defined by the stope optimizer.

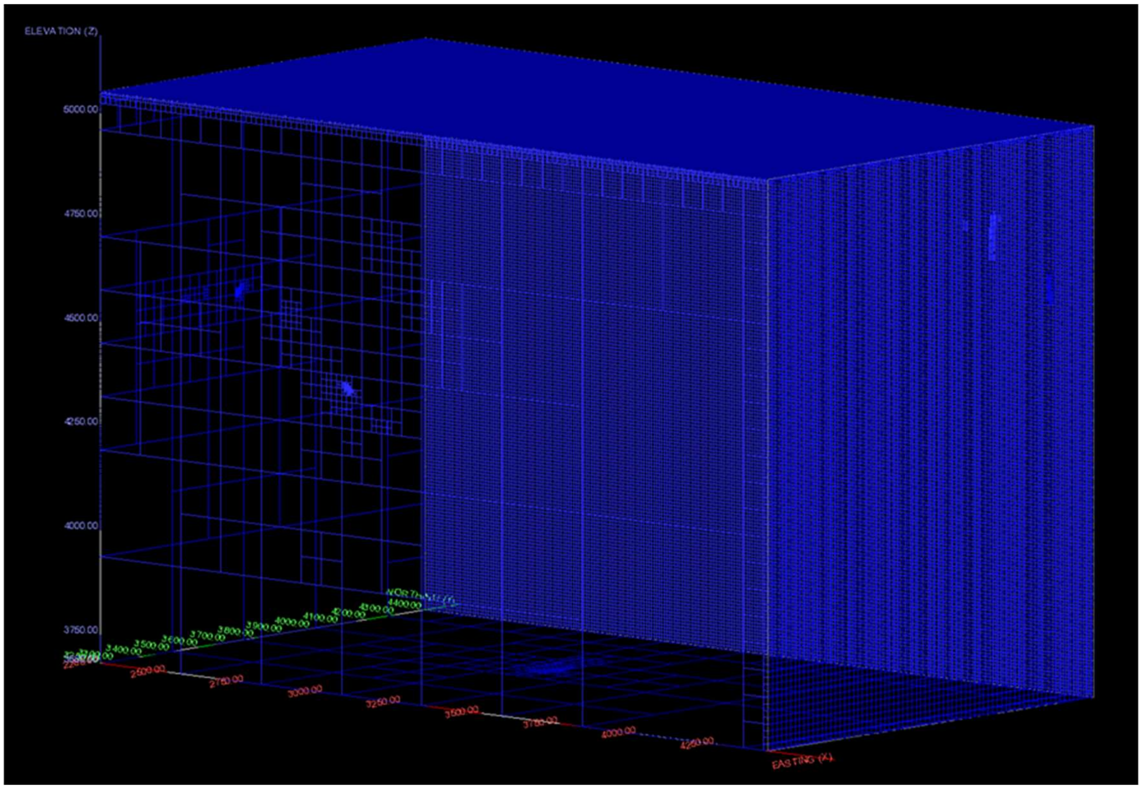


Figure 14.12 – Extent of the block model

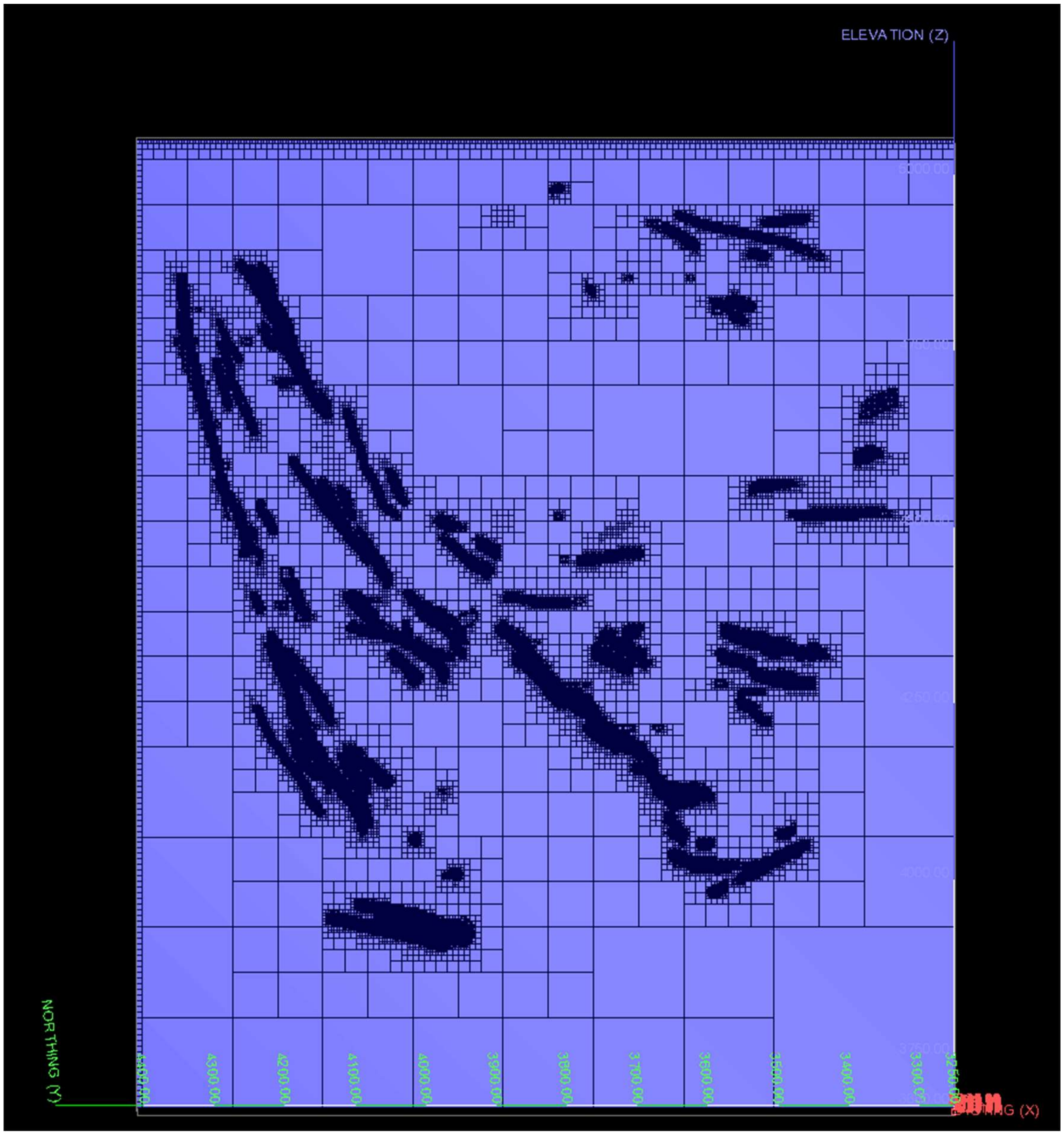


Figure 14.13 – Section of the block model looking west

At the scale of the Sleeping Giant deposit, this provides a reasonable block size for discerning grade distribution while still being large enough not to be misleading with respect to higher cut-off grade distributions within the model. The model was intersected with the crown pillar model to exclude blocks that extend above and cannot be retrieved without an updated geomechanical assessment.

Blocks not located within a mineralized solid but in close proximity were also interpolated, with the lower-grade composites remaining outside the interpreted solids. Those low-grade interpolated blocks were only considered as dilution during the stope optimizing procedure.

Blocks were classified into five types: blocks from (i) mineralized veins, (ii) bordering waste rock, (iii) previously mined block, (iv) overburden, and (v) surface pillar.

14.1.8 Grade block model

A grade model was interpolated using the 0.5 m capped composites (95 g/t Au) from conventional assay grade data. The interpolation method retained for the final resource estimation was ID2 with capping of high-grade values. The ID2 method was preferred because this deposit includes many high-grade gold values that locally create high-grade pockets of gold. The ordinary kriging (OK) interpolation method tends to smooth the grade and therefore minimizes the impact of these high-grade pockets while giving higher grades to blocks elsewhere.

14.1.9 Dilution envelope

During the stope optimization procedure with Deswik.SO, the waste rock adjacent to the mineralized veins is considered dilution. To determine the grade of this dilution, 5-m buffers were created around every mineralized vein. Composites (0.5 m) within those buffers have been created in the waste rock, excluding the mineralized veins, to ensure that only waste grades are interpolated. Because continuity was difficult to determine from very low-grade waste composites, the ID2 interpolation method was used for waste. The ellipsoid used was spherical, with a radius of 30 m. The waste composite data set was capped at 2 g/t Au (Figure 14.13). The data set contains 489,651 composites, but only 298 composites were capped at 2 g/t Au, representing 0.06% of the set. Only one pass has been used to interpolate waste blocks (Table 14.3).

Table 14.3 – Mining dilution estimation settings

	Minimum # of composites	Maximum # of composites	Maximum Composites/DDH	Minimum # of DDH
1st pass	1	10	2	1

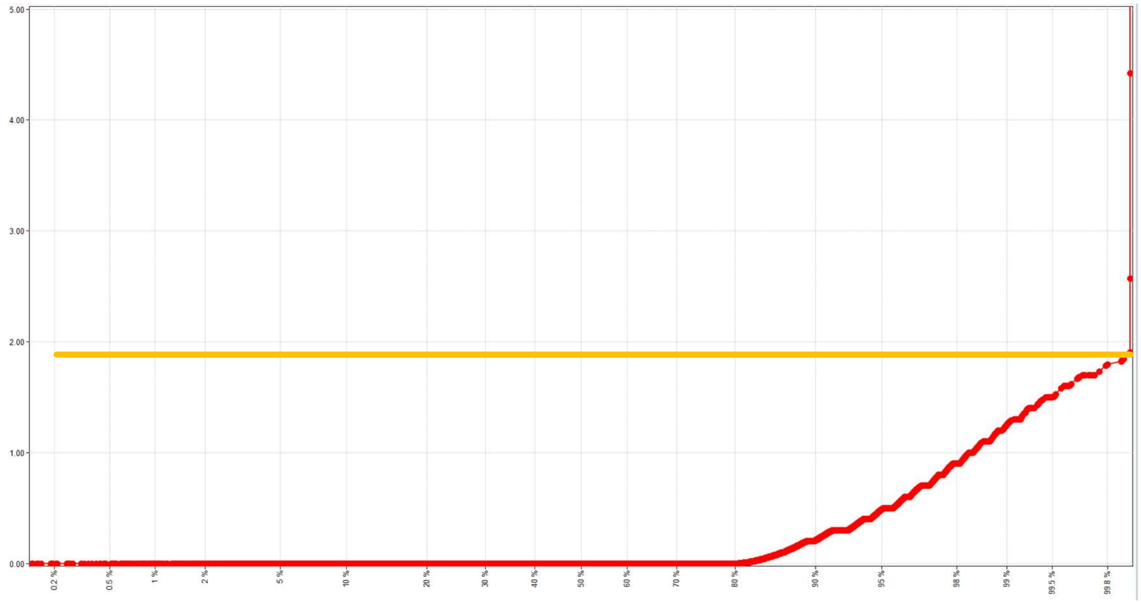


Figure 14.14 – Frequency plot of waste composites with capping at 2 g/t Au

14.1.10 Estimation settings

Two domains can be observed in the deposit, separated by the NW-SE fault (Figure 14.14), which dips 56° towards 200°. The section of the deposit north of the NW-SE fault has a grade continuity of approximately 120 m. The maximum range (grade continuity) was determined by variography. The section south of the NW-SE fault has a grade continuity of approximately 150 m.

ID2 was the interpolation method selected to estimate the blocks in the deposit.

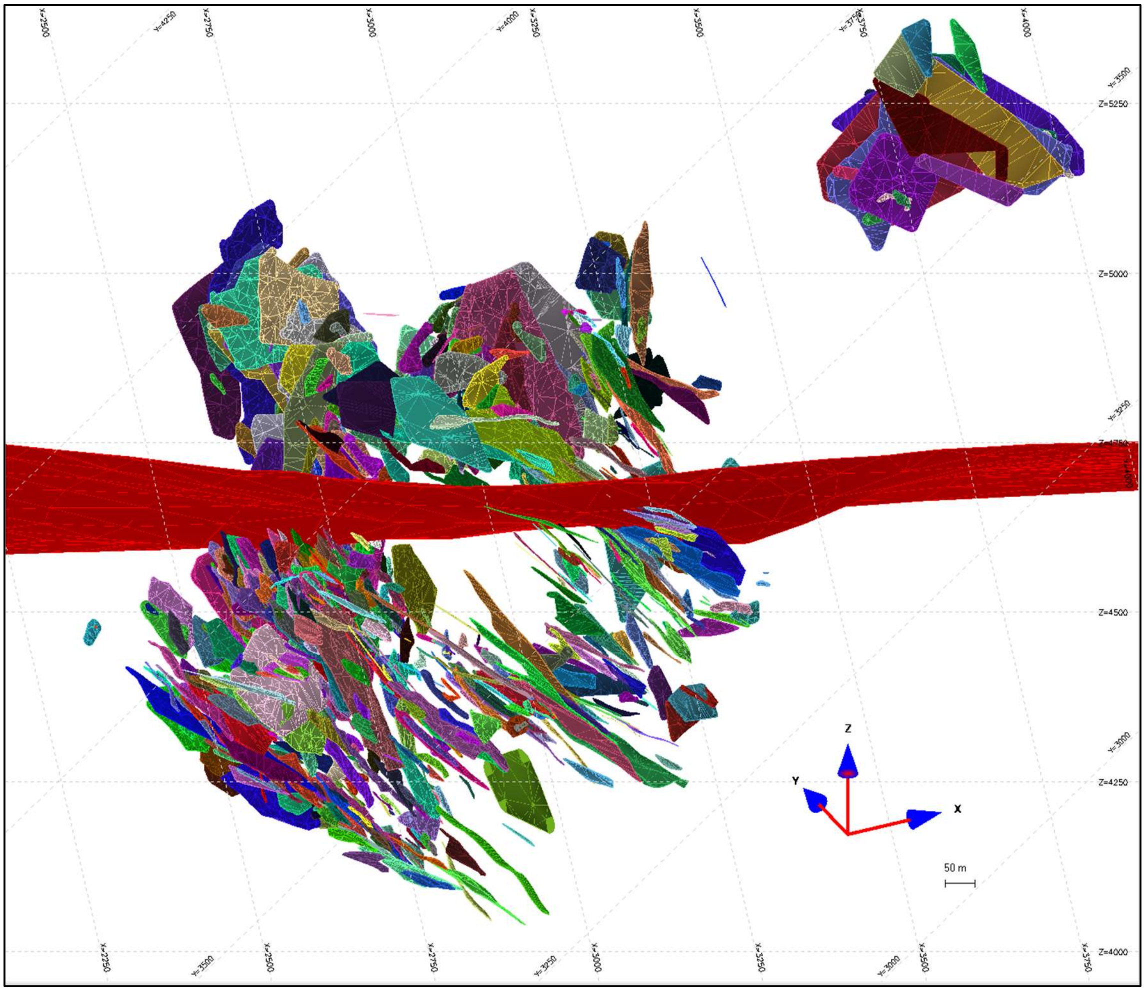


Figure 14.15 – Oblique view of the NW-SE fault (red)

Three passes were used to interpolate the grade of all blocks in the grade shells; however, the ellipsoid size is different to the north and south of the fault.

North of the fault, for Pass 1, the search ellipse size for all vein domains was set at 30 m x 30 m x 10 m in the X, Y, and Z directions; for Pass 2, the search ellipse size for each domain was set at 60 m x 60 m x 20 m; for Pass 3 the search ellipse size was set at 120 m x 120 m x 20 m (Table 14.4).

Table 14.4 – Northern section ellipsoid sizes

	Long axis	Medium axis	Short axis
1st pass	30m	30m	10m
2nd pass	60m	60m	20m
3rd pass	120m	120m	20m

South of the fault, for Pass 1, the search ellipse size for all vein domains was set at 30 m x 30 m x 10 m in the X, Y, and Z directions; for Pass 2, the search ellipse size for each domain was set at 75 m x 75 m x 20 m; for Pass 3 the search ellipse size was set at 150 m x 150 m x 20 m (Table 14.5).

Table 14.5 – Southern section ellipsoid sizes

	Long axis	Medium axis	Short axis
1st pass	30m	30m	10m
2nd pass	75m	75m	20m
3rd pass	150m	150m	20m

Blocks were classified as Indicated if they were populated with grades during Pass 1 and 2 of the interpolation procedure. Blocks were classified as Inferred if populated with grades during Pass 3 of the interpolation procedure.

Grades were interpolated into blocks using a minimum of 7 and maximum of 20 composites to generate block grades during Pass 1; a minimum of 5 and maximum of 20 composites to generate block grades during Pass 2 (maximum of 2 sample composites per drill hole); and a minimum of 1 and maximum of 20 composites to generate block grades during pass 3 (Table 14.6). Each vein is estimated individually with its own set of composites. Each vein has an ellipsoid with its own best-fit orientation.

Table 14.6 – Estimation setting

	Minimum # of composites	Maximum # of composites	Maximum Composites/DDH	Minimum # of DDH
1st pass	7	20	2	4
2nd pass	5	20	2	3
3rd pass	1	20	2	1

14.1.11 Economic parameters and Cut-off grade

Cut-off grade (CoG) parameters were determined by QP Eric Lecomte using the parameters presented in Table 14.7 and Table 14.8. The deposit is reported at a rounded CoG of 4.25 g/t Au using the long-hole (LH) mining method and 5.00 g/t Au using the room & pillar (R&P) mining method. The choice between the mining methods depended on the stope angle given by Deswik Mineable Shape Optimizer (DSO) run, using the LH method for all zones where the general dip is greater or equal to 43° and the R & P method for all zones where the general dip is less than 43°.

The QP considers the selected CoG of 4.25 g/t Au and 5.00 g/t Au to be adequate based on the current knowledge of the Project. The CoG are considered to be instrumental in outlining mineral resources with reasonable prospects for eventual economic extraction for an underground mining scenario.

Table 14.7 – Input Parameters Used to Calculate the Underground Cut-off Grade (using the LH Mining Method) for the Sleeping Giant Project

Input parameter	Value
Gold price (US$/oz)	1,650
Exchange rate (USD:CAD)	1.30
Gold Price ($/oz)	2,145
Royalty (%)	2.00
Recovery (%)	95
LH minimum stope angle (°)	43
Global mining costs ($/t)	213.96
Processing & transport costs ($/t)	35.10
G&A costs ($/t)	22.09
Total cost ($/t)	271.16
Mineral resource cut-off grade (g/t Au)	4.25

Table 14.8 – Input Parameters Used to Calculate the Underground Cut-off Grade (Using the R&P Mining Method) for the Sleeping Giant Project

Input parameter	Value
Gold price (US$/oz)	1,650
Exchange rate (USD:CAD)	1.30
Gold Price ($/oz)	2,145
Royalty (%)	2.00
Recovery (%)	95
Global mining costs ($/t)	261.56
Processing & transport costs ($/t)	35.10
G&A costs ($/t)	22.09
Total cost ($/t)	318.75
Mineral resource cut-off grade (g/t Au)	5.00

For the LH method, the DSO parameters used a standard length of 5.0 m longitudinally along the strike of the deposit, a height of 15.0 m and a minimum width of 1.7 m. The minimum shape measures 5.0 m x 7.5 m x 1.7 m. The standard shape was optimized first. If it was not potentially economic, smaller stope shapes were optimized until they reached the minimum mining shape.

For the R&P method, the DSO parameters used a standard length of 5.0 m longitudinally along the strike of the deposit, a height of 15.0 m along dip and a minimum width of 1.7 m. The minimum shape measures 5.0 m x 7.5 m x 1.7 m. The standard shape was optimized first. If it was not potentially economic, smaller stope shapes were optimized until they reached the minimum mining shape.

The use of the conceptual mining shapes as constraints to report mineral resource estimates demonstrates that the criterion of "reasonable prospects for eventual economic extraction" has been met, as defined in CIM Guidelines (2019).

14.1.12 Mined out exclusion

Many stopes have been mined out during the previous years. Abcourt provided InnovExplo with 3D solids of the infrastructure (shaft, ramp, drifts) and many stopes (Figure 14.16). Older stopes have not been generated in 3D. InnovExplo had to use a longitudinal view of stopes (not provided in 3D solid format) to establish the correspondence between depleted areas and the current 3D interpretation (Figure 14.17 and Figure 14.18). A 5-m buffer was applied to depleted stopes and 2.5 m to infrastructure to ensure that all previously mined areas were excluded. All blocks within those boundaries were set to zero for mineral resource reporting.

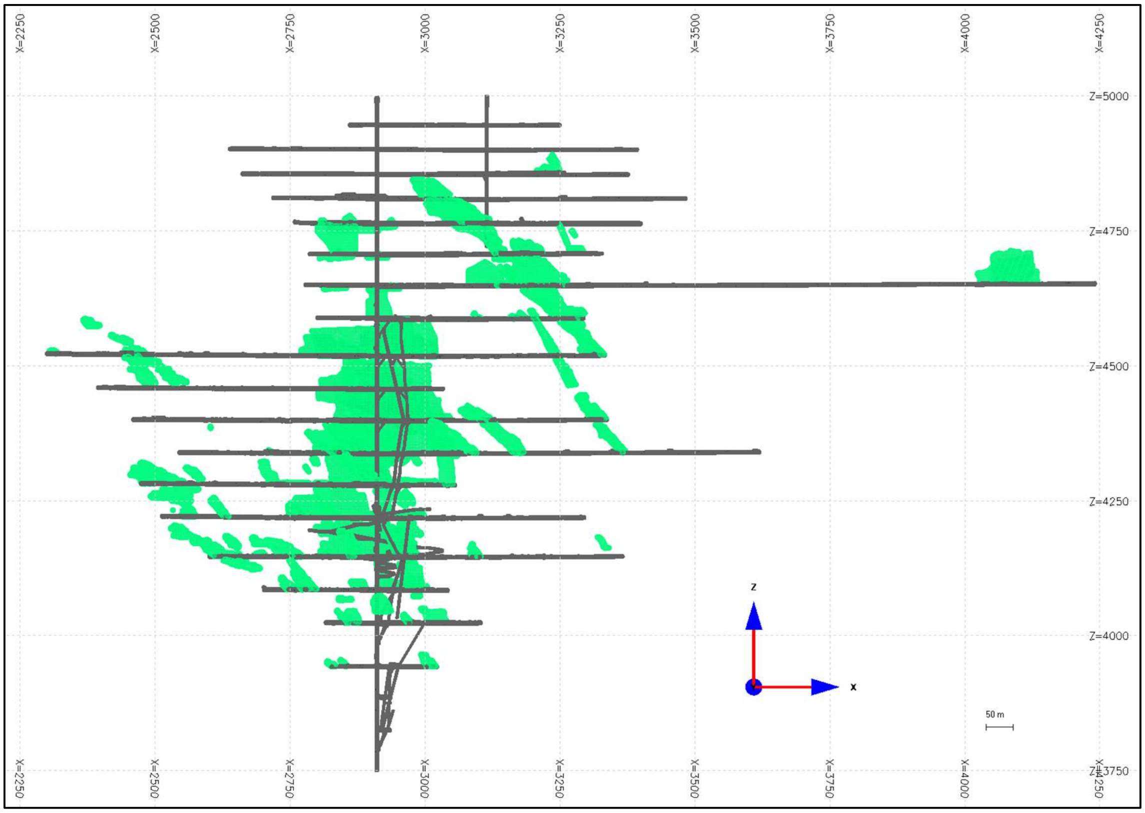


Figure 14.16 – 3D depletion solids

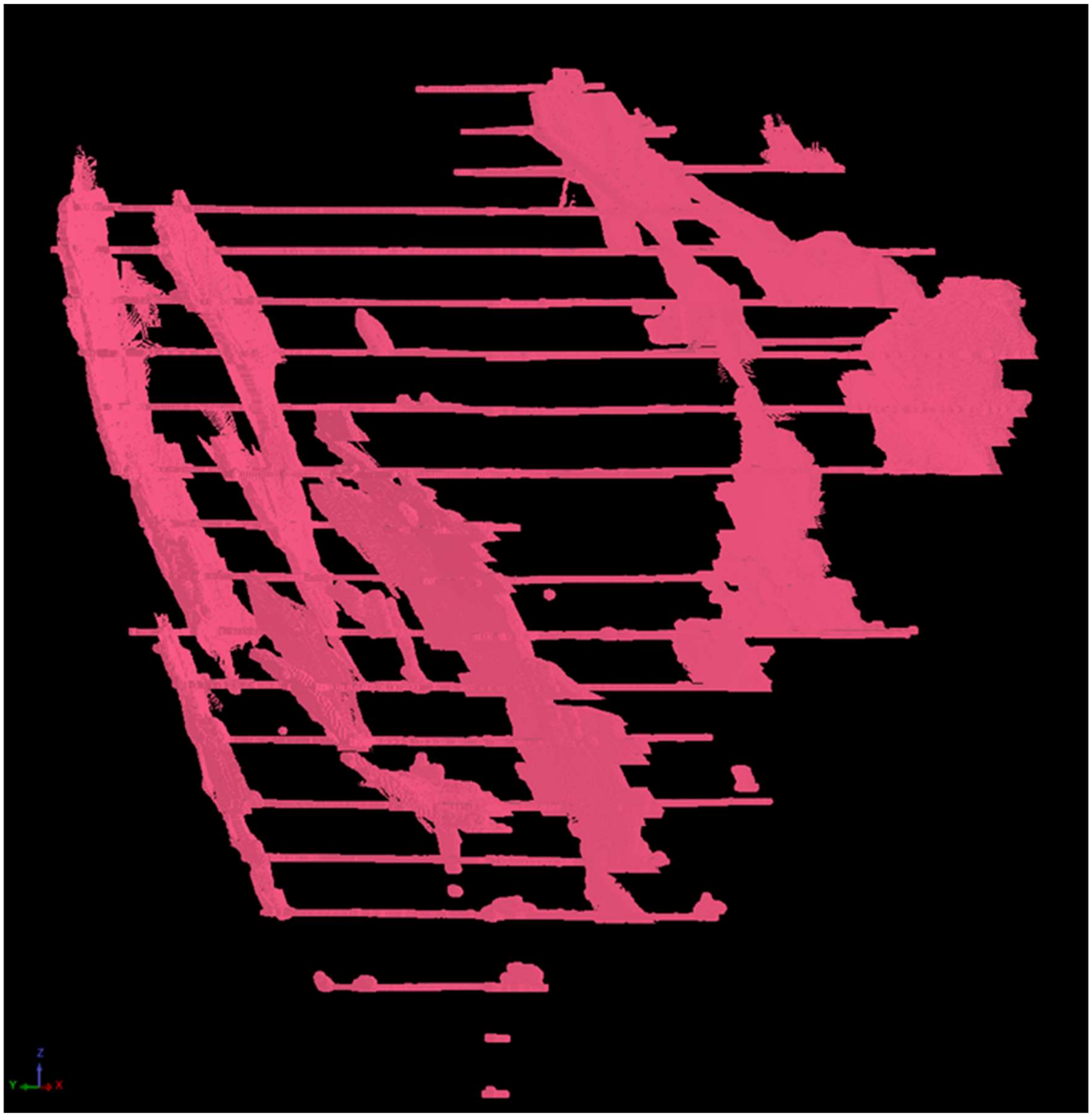


Figure 14.17 – Final depleted block looking S-W

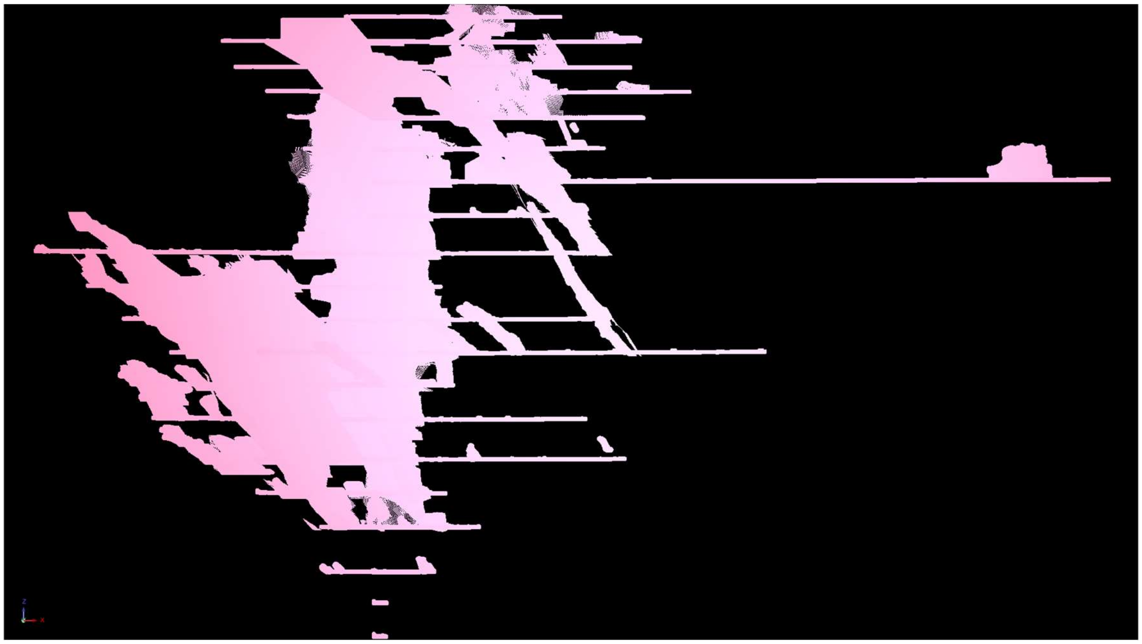


Figure 14.18 – Final depleted block looking west

14.1.13 Overburden and crown pillar

The overburden surface was created using the described lithological intervals. This surface was then translated 50 m deeper to create the surface crown pillar bottom limit (Figure 14.19). No blocks above the surface crown pillar limit have been included in the 2022 MRE.

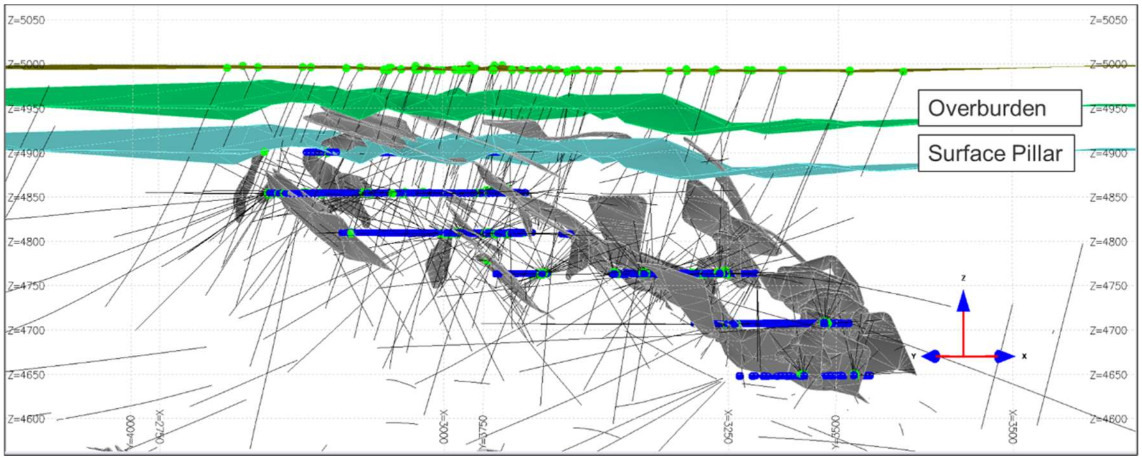


Figure 14.19 – Overburden and Surface Pillar (section view)

14.2 Mineral Resource Classification, Category or Definition

The resource classification definitions used for this report are those published by the Canadian Institute of Mining, Metallurgy and Petroleum in their 2019 document "*CIM Standards on Mineral Resources and Reserves – Definitions and Guidelines*" ("CIM Definition Standards") and defined as follows:

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.

14.2.1 Geological resource classification

Blocks have been classified consequently with the pass number at which they have been estimated. No block has been classified as measured because of the use of the buffers around the existing stopes and infrastructures. Blocks interpolated in the first pass were classified as Indicated. Blocks interpolated in the second pass have been classified as Inferred. Blocks interpolated in the third pass have been classified as Potential (Figure 14.20).

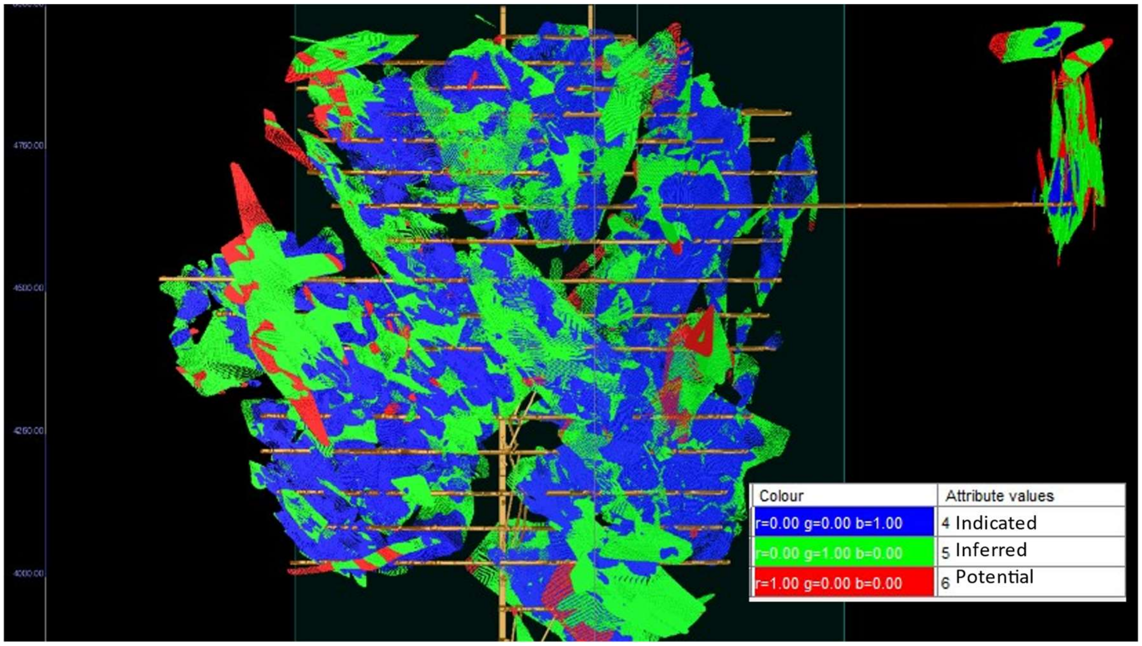


Figure 14.20 – Block classification

14.2.2 Mineral resource selection from stope optimizer

To ensure potentially mineable resources following CIM Definition Standards (2019), potential stope shapes were optimized using the Deswik Stope Optimizer (DSO) from the software Deswik. The block model used was generated after completing the aforementioned geological estimation and classification (Figure 14.21). This allowed for more flexibility during the optimization process, including sub-shapes and anneal parameters to ensure maximum resource conversion to DSO. The additional parameters used for the optimization process are summarized in Table 14.9.

Table 14.9 – DSO parameters

		Mining Method	
Parameters	**Units**	**Long-hole**	**Room & pillar**
Cut-Off Grade	g/t	4.25	5.00
Level (Height)	m	15	15
Section (Length)	m	5	5
Stope Width (Min)	m	1.7	1.7
Side Ratio	N/A	5	5
Dip (Min/Max)	Deg	43/90	0/43

Regarding the DSO-based resource classification, the dominant system is used to ensure all resources are associated with one of the evaluated categories (i.e., Measured, Indicated, or Inferred). The category of each DSO is dictated by the most prominent category by volume included in each solid.

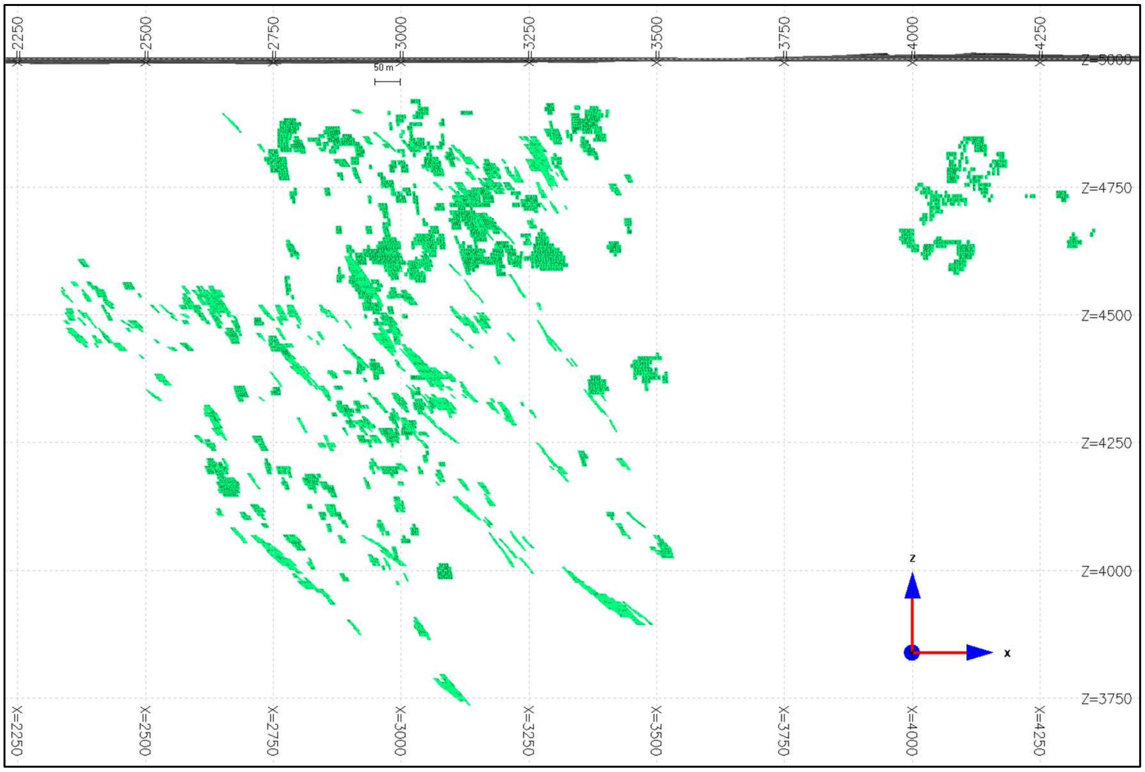


Figure 14.21 – Potential stope optimization looking north

14.3 Mineral Resource Estimation

The 2022 MRE (Table 14.11) includes all blocks ("must take blocks") that fall within a potentially mineable shape to satisfy the "reasonable prospects for eventual economic extraction" as specified by the CIM Definition Standards. These guidelines were introduced after the 2019 MRE for Sleeping Giant was calculated.

Table 14.10 – Mineral Resource Estimate for the Sleeping Giant Mine (Effective as of December 12, 2022)

Potential Long Hole (cut off at 4.25 g/t Au)			Potential Room and Pillar (cut-off at 5.0 g/t Au)			Total		
Tonnes	**Grade (Au g/t)**	**Ounces Au**	**Tonnes**	**Grade (Au g/t)**	**Ounces Au**	**Tonnes**	**Grade (Au g/t)**	**Ounces Au**
Indicated Resources								
677,000	7.03	153,000	78,000	7.98	20,000	755,000	7.14	173,300
Inferred Resources								
677,000	8.13	177,000	207,000	10.67	71,000	884,000	8.74	248,300

Notes to the 2022 MRE:

1. The independent and qualified persons for the 2022 MRE, as defined by NI 43-101, are Olivier Vadnais-Leblanc (P.Geo.) and Eric Lecomte (P.Eng.), both from InnovExplo Inc.
2. These mineral resources are not mineral reserves because they do not have demonstrated economic viability. The results are presented undiluted and are considered to have reasonable prospects of economic viability. The 2022 MRE follows CIM Definition Standards (2014) and CIM Guidelines (2019).
3. The estimate encompasses 846 mineralized zones modelled using a minimum geological width of 0.5 m in Genesis software.
4. A density value of 2.85 g/cm3 (based on measurements and mine and mill reconciliation) was assigned to all mineralized zones.
5. High-grade capping supported by statistical analysis was established at 95 g/t Au for all mineralized zones and applied to the composite data. Composites (0.5 m) were calculated within the zones using the grade of the adjacent material when assayed or a value of zero when not assayed.
6. The RPEEE requirement is fulfilled using cut-off grades based on reasonable mining parameters, locally constrained within DSO shapes using a minimum mining width of 1.7 m for both potential methods. It is reported at a rounded cut-off grade of 4.25 g/t Au using the long-hole ("LH") method and 5.0 g/t Au using the room and pillar ("R&P") method. The cut-off grades were calculated using the following parameters: mining cost = C$213.96/t (LH) to C$261.56/t (R&P); processing cost = C$35.10/t; G&A = C$22.09/t; gold price = US$1,650.00/oz and USD:CAD exchange rate = 1.30. The cut-off grades should be re-evaluated in light of future prevailing market conditions (metal prices, exchange rates, mining costs etc.).
7. The estimate was completed using a sub-block model in Surpac 2022. A 4m x 4m x 4m parent block size was used (1m x 1m x 1m sub-blocked). Grade interpolation was obtained by ID2 using hard boundaries.
8. The mineral resource estimate is classified as Indicated and Inferred. The Inferred category is defined with a minimum of three (3) drill holes within the areas where the drill spacing is less than 75 m and shows reasonable geological and grade continuity. The Indicated mineral resource category is defined with a minimum of four (4) drill holes within the areas where the drill spacing is less than 30 m and shows reasonable geological and grade continuity.
9. The number of metric tons was rounded to the nearest hundred, following the recommendations in NI 43-101, and any discrepancies in the totals are due to rounding effects. The metal contents are presented in troy ounces (tonnes x grade / 31.10348) rounded to the nearest hundred.
10. The independent and qualified persons for the 2022 MRE are not aware of any known environmental, permitting, legal, political, title-related, taxation, socio-political, or marketing issues that could materially affect the estimate.

14.3.1 Block model validation

Validation was done visually and statistically by the QP to ensure that the final mineral resource block model was consistent with the primary data.

Block model grades, composite grades and assays were visually compared on sections, plans and longitudinal views for both densely and sparsely drilled areas. No significant differences were observed. A generally good match was noted in the grade distribution without excessive smoothing in the block model.

The grade-tonnage curve of the deposit (Figure 14.22) is also a good indicator of grade interpolation. The smooth grade curve reflects good handling of the interpolation and the absence of high-grade blocks.

The trend and local variation of the estimated ID2 and OK models were compared to the nearest-neighbour (NN) model and composite data using swath plots in an east-west direction for the Indicated and Inferred blocks (Figure 14.23). Cases in which the composite mean is higher than the block-mean are often a consequence of clustered drilling patterns in high-grade areas. It is also worth noting that the mean of the composites is independent of the classification.

The comparison between composite and block grade distribution and the overall validation did not identify any significant issues.

Figure 14.22 – Grade-tonnage curve

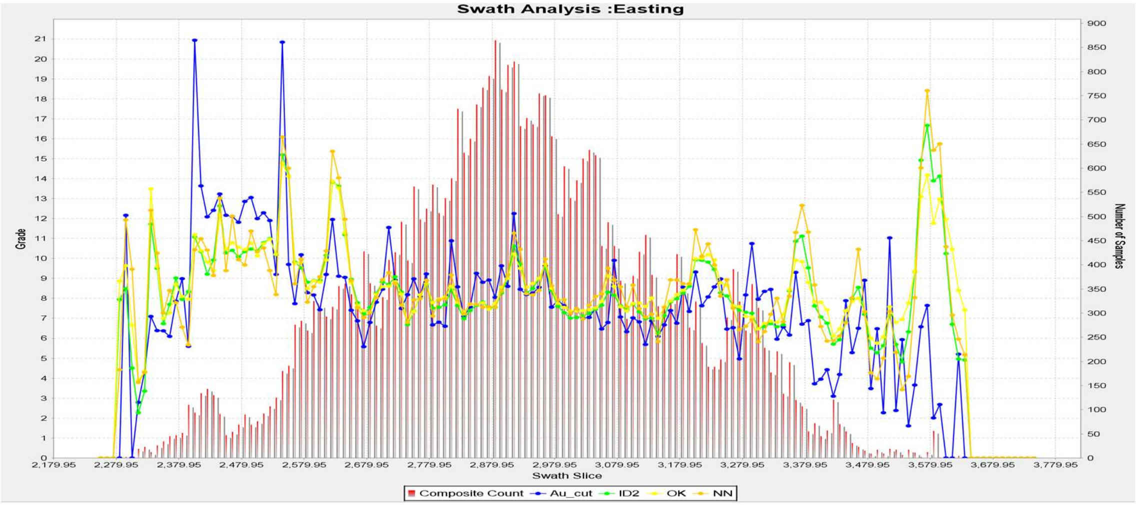


Figure 14.23 – Swath plot analysis

14.3.2 Sensitivity to Cut-off grade

Table 14.11 shows the cut-off grade sensitivity analysis of the 2022 MRE. The reader should be cautioned that the numbers provided should not be interpreted as a mineral resource statement. The reported quantities and grade at different cut-off grades are presented in-situ and for the sole purpose of demonstrating the sensitivity of the mineral resource model to the selection of a reporting cut-off grade.

Table 14.11 – Sensitivity of the 2022 MRE to Different Gold Prices (Effective Date of December 12, 2022)

Gold Price US$	Cut-Off grades LH/R&P (g/t Au)	Tonnes	Grade (Au g/t)	Troy Ounces (Oz Au)
Indicated Resources				
1,475 US$	4.75/5.60	621,000	7.73	154,200
1,550 US$	4.55/5.30	675,000	7.51	163,000
1,650 US$	4.25/5.00	755,000	7.14	173,300
1,750 US$	4.00/4.70	826,000	6.89	183,000
1,825 US$	3.85/4.50	871,000	6.72	188,300
Inferred Resources				
1,475 US$	4.75/5.60	754,000	9.43	228,600
1,550 US$	4.55/5.30	812,000	9.13	238.400
1,650 US$	4.25/5.00	884,000	8.74	248,300
1,750 US$	4.00/4.70	949,000	8.40	256,200
1,825 US$	3.85/4.50	992,000	8.20	261,400

Note: Numbers may not add up due to rounding. The reader is cautioned that the figures provided in Table 14.11 should not be interpreted as a statement of mineral resources. Quantities and estimated grades for different gold prices (and cut-off grades) are presented for the sole purpose of demonstrating the sensitivity of the mineral resources model to the choice of a specific gold price.

15. MINERAL RESERVE ESTIMATES

Not applicable at the current stage of the Project.

16. MINING METHODS

16.1 Introduction

The Sleeping Giant mine was a conventional operation that suspended is production in 2014. Since then, the underground facilities have been maintained under care and maintenance and most of the underground excavations are still accessible. The Sleeping Giant project ("The project") is considering production using similar methods to those used in previous operations, with a combination of long-hole ("LH"), room-and-pillar ("R&P") and shrinkage mining methods. The selection of the different methods has been determined based on the physical characteristics of the mineralisation. It is planned, in the first instance, to begin underground work by rehabilitating the shaft service compartment and the escapeway. Subsequently, depending on the priorities of the production plan, rehabilitation of the existing galleries will continue over the following months on the different levels. Mined material will be processed at a rate of approximately 350 tpd

The proposed underground operations at Sleeping Giant would utilize conventional mining methods, using track drift to haul mineralized material to the existing ore-pass system. To reach the optimized stopes, new drifts are excavated from existing levels. Additionally, some new levels will be developed near the bottom of the shaft (below N-1060). The proposed operation has an expected mine life close to six (6) years following the pre-production period.

The Project will involve an underground operation with a proposed mining method optimized to the deposit geometry (narrow veins) and previously used production techniques. It will employ a variety of stoping methods (LH, R&P, and shrinkage). Mining voids will not be backfilled, continuing the previous approach.

The optimization process has been completed using Deswik™ DSO (stope optimizer module), whereas the planification has been optimized using the Deswik™ Scheduler. Both programs ensure optimal and extensive outputs to be used in the final cashflow. Though not required at this phase of the project, this adds a significant value to the study and minimizes risks associated with a simpler approach.

The two main objectives of the current assessment and schedule is to minimize the production ramp-up and to maximize resulting grade.

16.1.1 Cut-off grade calculation

A cut-off grade has been calculated to evaluate the mineralized material that has an economic potential. Parameters for the cut-off calculation are summarized in Table 16.1 for the three mining methods. The primary differences between the three methods are the dilution and the mining cost.

Table 16.1 – Cut-off Grade Parameters

Parameters	Unit	Long hole	R&P	Shrinkage
Gold Price	US$/oz	1,650	1,650	1,650
Exchange Rate	CAD:USD	1.00:1.30	1.00:1.30	1.00:1.30
Royalty	%	2.0	2.0	2.0
Refining Cost	$/oz	5.00	5.00	5.00
Processing Cost	$/t treated	35.10	35.10	35.10
Metallurgical Recovery	%	95.0	95.0	95.0
Mining Recovery	%	95.0	95.0	95.0
Mining Dilution	%	17.6	23.5	2.9
Mining Cost	$/t treated	214.35	261.94	249.08
General and Administration	$/t treated	21.70	21.70	21.70
Cut-Off Grade	**g/t**	**4.50**	**5.25**	**5.05**

16.1.2 Economic mineralized material

Following the optimization process, further filtering of the optimized stopes resulted in the final economical mineralized material. Sorting of the stopes is based on block value and required development. Including dilution, recovery and following the planification process, the final resource with economic potential is presented in Table 16.2.

Table 16.2 – Economic mineralized material summary

Mining method	Indicated resources			Inferred resources		
	Tonne (t)	Grade (g/t)	Troy Ounces (oz)	Tonne (t)	Grade (g/t)	Troy Ounces (oz)
Long hole	57,000	6,08	11,100	140,200	7.69	34,700
Room & Pillars	161,600	6,19	32,200	147,400	9.01	42,700
Shrinkage	59,000	7,64	14,500	155,100	10.51	52,400
Total	277,600	6,48	57,800	442,700	9.12	129,800

16.2 Geotechnical Considerations

16.2.1 Historic Practices

Historically, the mine was exploited using different mining methods. Between 2009 and 2022, the principal selected mining methods were shrinkage, and R&P. Also, there were a few stopes that were exploited with the long-hole method.

The principal method that was prioritized since the activities re-started in 2023 was long-hole stopping with a dip above 40°. The stopes with shallower dips will be exploited using the R&P method.

16.2.2 RQD (Rock Quality Designation)

The geology department from the Sleeping Giant operations systematically collected RQD data from all drill core.

In general, the bedrock quality at the Sleeping Giant mine is good to very good (RQD of 70 – 90%). The RQD data from different depths of the mine are presented in Table 16.3.

Table 16.3 – RQD at Sleeping Giant

Zone	RQD	Method/Source
Level 145 Zone 5	70 – 80 %	Drilling cores
Level 415 – 605 Zone 20	85 %	RQD = 115 – 3.3 Jv (Doucet et al., 1996)
Level 545 – 605 Zone 8	85 – 90 %	R. Royer (Iamgold) in CANMET (2004)
Level 785 - 975	70 – 90 %	Drilling cores

16.2.3 Surface Pillar

A report on the stability of the surface pillar was produced by the Sleeping Giant department of Engineering in 1993. The report confirms the stability of the surface pillar for the stopes already exploited. Also, Sleeping Giant mine is not under the influence of the water table (Journaux, Bédard & assoc. Inc, 2001). Thus, the mine does not need conformity of Articles 76, 77, 79 and 80 of the Regulation Respecting Occupational Health and Safety in Mines. The old stopes that were already exploited were located at 50m below surface meaning that the statement above also applies to the new stopes and that the crown pillar should be stable on long term.

16.2.4 Rock Compressive Strength

The main type of rock subjected to stress changes is the massive andesite. Lab tests were performed in 1996 on 22 samples of massive and pillow andesite by Laval University. The data were interpreted by CANMET in 2004 with the goal to determine the Hoek-Brown and Mohr Coulomb failure criteria for andesite. The same properties were attributed to massive and pillow andesite. Table 16.4 shows the mechanical proprieties extracted from the 2022 Sleeping Giant Ground Control Campaign.

Table 16.4 – Mechanical Proprieties of Andesite

UCS (Uniaxial Compressive Strength)		145 MPa
Hoek - Brown	mi (intact)	20
	s	0.021
Mohr Coulomb (Intact)	c (cohesion)	11 MPa
	Φ (friction angle)	40°
	Cres (residual cohesion)	7 MPa
	Φres (residual friction angle)	35°
v (Poisson coefficient)		0.25
Dil (Dilatation angle)		5°
ERM (In-situ Young's Modulus)		35000 MPa
σt (Intact tensive strength)		-0.5 MPa

16.2.5 Rock Mass Characterization

Where a mineralized zone is located in a layer of laminated sediments between two andesite layers, the behaviour of the rock mass at the back of the excavation depends on the lithology. A relatively smooth back would yield when the contact between the laminated zone and the andesite is exposed meaning that the laminated zone is totally mined until contact with the andesite. However, when the mineralization does not go to the contact, it is often hard to retain the laminated zone between the back of the excavation and the andesite contact. For the purpose of this study, no distinction was made between the two scenarios as it is assumed that the additional dilution will be mostly ore.

The formation of Andesite with quartz-carbonate veinlets (1 to 5 mm thick) can weaken the bedrock depending on the stress conditions.

Discontinuity surveys were conducted in zones 20, 30 and 40 by Doucet et al. in 1996 and in zone 8 by CANMET in 2004.

Different rock mass classifications were realized at Sleeping Giant in zones 20 and 30. Those geomechanical classifications helped determine the orientation of the principal families of joints to define the quality of the bedrock. Those classifications define the different zones in the bedrock that can be considered as unique geomechanical domains. The differences between each zone are not significant.

Barton's classification from the Norwegian Geotechnical Institute (NGI) (Barton et al., 1974) and the Geological Strength Index (GSI) were the main geomechanical classification use for the project. The quality index ("Q") is calculated as followed:

$$Q = \frac{RQD}{J_n} * \frac{J_r}{J_a} * \frac{J_w}{SRF}$$

RQD = Rock Quality Designation

J_n = Parameter for the number of joint families

J_r = Parameter for the roughness of the discontinuities

J_a = Parameter for the weathering of the discontinuities

J_w = Parameter for the consideration of water

SRF = Parameter for in-situ stress

Table 16.5 summarized the values that were obtained until November 2022 from the various surveys achieved at Sleeping Giant by the site and us for the stope stability calculation. For the quality index Q, J_w and SRF are given a value of 1, which represents the conditions observed during the ground control campaign of 2022. For the Sleeping Giant mine, J_w = 1 and SRF = 0.5 – 2 are judged representative of the site condition depending on the induced stress.

Table 16.5 – Geomechanical classification

Parameter	Hanging Wall			Comments
	Low	High	Typical	
RQD	70	85	80	Estimated visually in the stopes
J_n	12	9	12	3 families + random
J_r	1	2	1	Planar and smooth joints
J_a	1	0.75	0.75	Closed joints with quartz-carbonate filling
Q'	5.8	25.2	8.9	

The geological strength index (GSI) was also found for the Andesite. According to the ground control campaign done by Abcourt in November 2022, the GSI for Andesite was found to be 65.

16.2.6 In-situ Stress

No in-situ stress measurements have been conducted at the Sleeping Giant mine. In 2004, CANMET did a retro-analysis with the software Map3D to consider an alternative tensor. In this tensor, the major stress is oriented at 045°, which is perpendicular to the last structurally associated event; in this case the intrusion of the felsic dykes. Following this retro-analysis, this tensor was kept and used for all subsequent modelling. The tensor is presented in Table 16.6.

Table 16.6 – Evaluation of In-Situ Stress

Stress	Minor Principal Stress (σ_3, MPa)	Intermediate Principal Stress (σ_2, MPa)	Major Principal Stress (σ_3, MPa)
Intensity*	0.028D	0.034D	0.050D
Direction	000°	315°	045°
Dip	90°	0°	0°

*D = Depth in m

16.3 Ground control

16.3.1 Method for Opening Designs

The *Mathews-Potvin Method (Potvin, 1988)* is an empirical calculation technique to estimate the stability of an open excavation. The method considered multiple factors related to the rock quality and in situ condition to evaluate the stability of the stope. N' is found by multiplying Q' by the A, B and C factors, which take respectively into account the induced stress in the walls, the orientation of the critical discontinuity families for wall stability, and the possible failure mode in the wall.

This stability number N' is used with the graph in Figure 16.1 to determine stable dimensions for the hanging wall by obtaining a hydraulic radius R_h. The hydraulic radius is defined as the ratio of the area over the perimeter of the span. Below are the parameters used for the stability graph method (Table 16.7).

Table 16.7 – Parameters used for the stability graph method

Parameter	Hanging Wall		
	Low	High	Typical
Q'	5.8	25.2	8.9
A	1	1	1
B	0.3	0.3	0.3
C	5.9	5.9	5.9
N'	10.3	45	15.7

For the long hole method, the height of the stope was set to be 52 m. Thus, from the graph it can be determined that the free face of the stope can have widths from 17 m to 59 m with support. For the Sleeping Giant mine, the width was set to 10 m due to past experiences and to what is currently being achieve at the Sleeping Giant mine. Thus, the hydraulic radius is 4.2m which yields to a stable excavation.

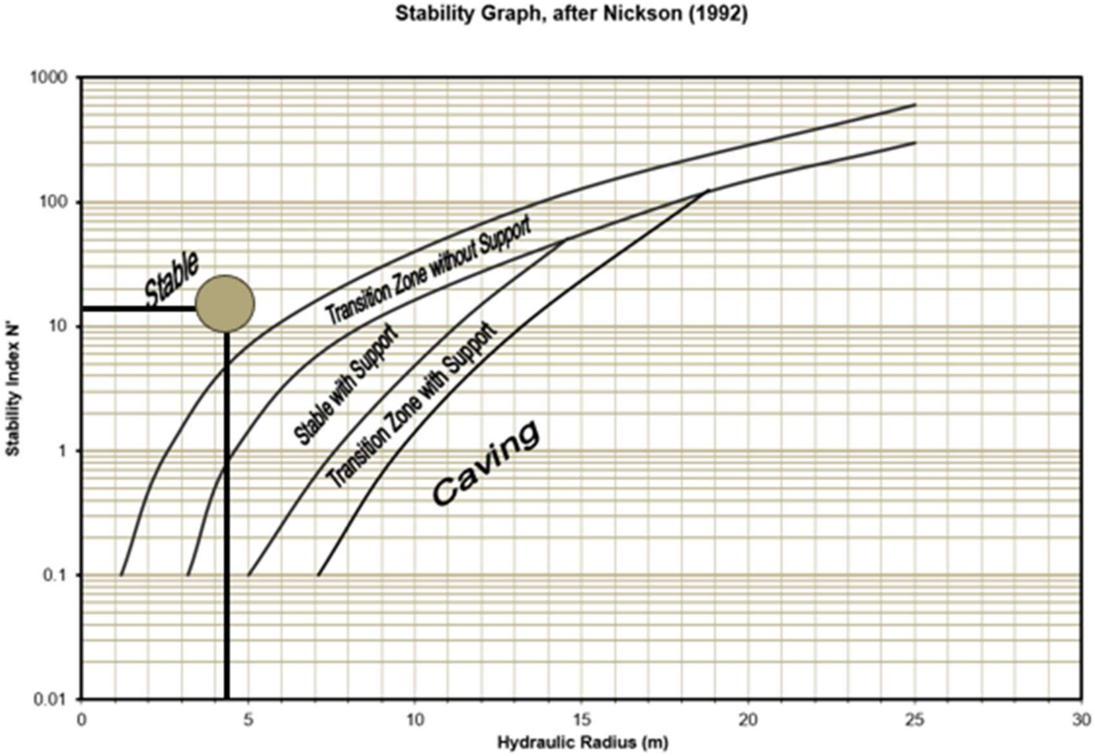


Figure 16.1 – Stability Graph for Sleeping Giant (Nickson, 1992)

The dilutions that were planned and used for the current study are found in the table below. The graph below confirms the dilutions that were chosen when the N' and the HR of the planned stopped dimensions are analysed. The ELOS should be smaller than 0.5m, and so are the dilutions used in the study. These dilution values were also confirmed by the experience of the mine site.

Table 16.8 – Dilutions used for long hole planning

	Long Hole
Dilution - HW	0.20m
Dilution - FW	0.10m

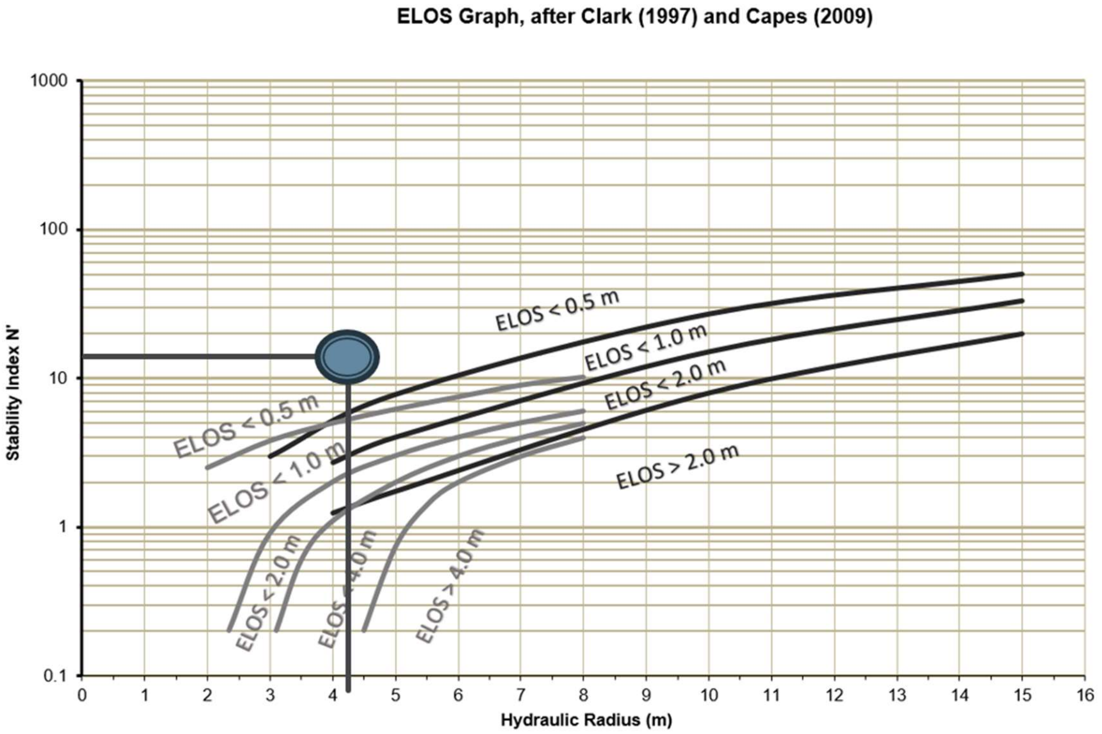


Figure 16.2 – ELOS graph for Sleeping Giant Clark (1997) and Capes (2009)

16.3.2 Evaluation of the dimensions of Room and Pillars

To determine the stress conditions within the pillar, the tributary area method was used. This method is very simple and commonly used for square pillars. The definition of the tributary area method proposes the following hypothesis:

- The stress is constant within the whole structure.
- Each pillar supports a rock column in which the area of the straight section is the sum of the pillar area and a portion of the excavation.
- The excavation area is distributed equally between the surrounding pillars.
- The force on each pillar is uniformly distributed on the whole surface and does not consider the stress in directions that are on different axis than the one from the pillar.

The pillar dimensions are 3 m x 3 m. Each pillar supports an area of 6m x 9m. The dilution planned in the room and pillar method is 0.30 m for the hanging wall and 0.10m for the footwall. Those dimensions were obtained from the Ground Control Campaign in November 2022 which considered the tributary area method to find those dimensions. These assumptions were also validated by the site experiences.

16.3.3 Sill pillar for long hole

For this study, a thickness of 2.5m was used for the sill pillars. This thickness came from past experiences, and from what is being currently achieved at Sleeping Giant.

16.3.4 Ground support

For all excavations except R&P, Table 16.9 should be followed. The length of the bolt is based on the Famer and Shelton rules, which state that L = 0.3B. The minimum spacing between the bolts can be based on the same set of rules stating that S = 0.5L or 3 to 4 times the dimension of the potential block.

Table 16.9 – Minimal bolt length needed

Width in m	Minimal Bolt Length Needed
Smaller than 5 m	5 ft
5 m – 7 m	7 ft
7 m – 10 m	10 ft

The geotechnical considerations (see Section 16.2) were used to evaluate the density of the ground support needed for the excavations to ensure a safe environment. The NGI method was chosen. With this method the excavation support ratio (ESR) needs to be employed. This factor is basically an inversely proportional safety factor. The ESR is used to find the equivalent width of the excavation in function of its utilisation. The ESR as well as the equivalent width are shown in Table 16.10.

Table 16.10 – Selection of the equivalent excavation width

Excavation	ESR	Maximal Width (m)	Equivalent Width (m)
Station	1.2	9	7.5
Garage	1.2	7	5.8
Refuge	1.2	4	3.3
Cross Cut	1.5	3	2.0
Haulage Drift	1.8	3	1.7
Draw Point	1.8	3	1.7
Sill	2.0	2	1.0
Working Place	3.0	3	1.0

Using the NGI chart (Figure 16.3), and a Q’ value of 6 (see Table 16.5), it is found that no additional support is needed except for station and the garage excavations, which will be systematically bolted with fibre reinforced sprayed concrete.

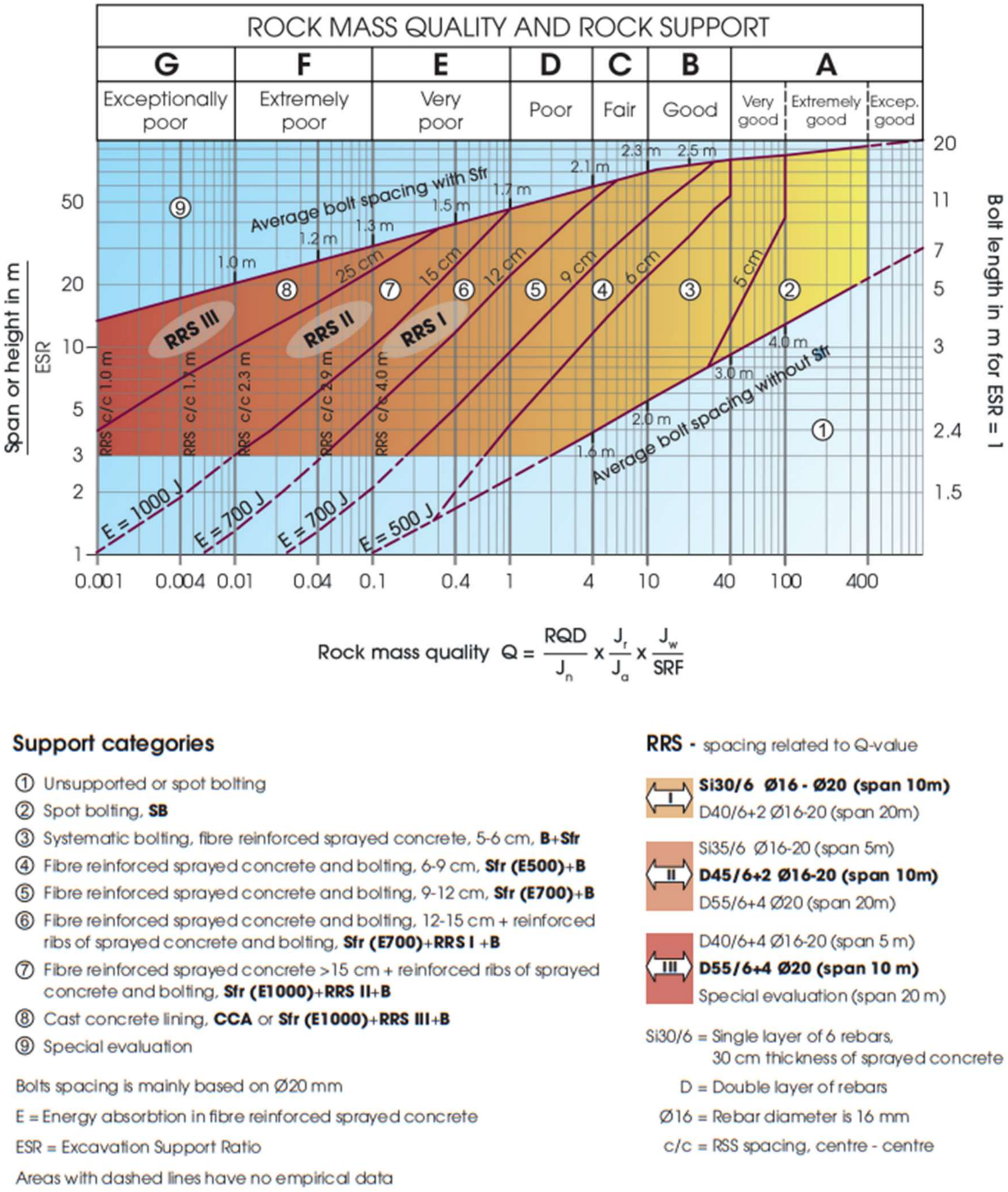


Figure 16.3 – NGI Chart for Rock Support (NGI, 2022)

The pressure on the back is calculated as follows:

$$P_{back} = \frac{2}{J_r^3\sqrt{Q}} = 0.1101\ MPa$$

Where: Q = 6 and J_r = 1.

The length of the bolts on the back is evaluated according to the NGI ground support chart elaborated by the Grimstad and Barton (1993) *Method.*

The minimal bolting density in drifts and the minimal spacing are calculated with the following equations:

$$D_{Min} = -0.227 * \ln(Q) + 0.893$$

$$S_{Max} = \sqrt{\frac{1}{D_{Min}}}$$

For Q = 6, D_{min} is found to be 0.49 bolt per m^2 of back, which corresponds to a maximal spacing of 1.43 m between each bolt.

It is important to notice that the Q index used here does not take into account the damages caused by blasting.

The main specification standards used for Sleeping Giant are presented in Table 16.11. Generally, mechanical bolts are the most commonly used, and resin and friction bolts are used in specific places during rehabilitation of openings. Those standards come from the 2022 Ground Control Campaign.

Table 16.11 – Bolting Specifications and Codes

D1 =	5’ mechanical bolts with a 4’x4’ regular pattern
D2 =	5’ mechanical bolts with a 3’x3’ regular pattern.
D3 =	5’ mechanical bolts with a 4’x3’ regular pattern
D4 =	5’ mechanical bolts with a 8’x3’ offset pattern
D5 =	5’ mechanical bolts with a 3’x3’ offset pattern
D6 =	7’ mechanical bolts with a 3’x3’ offset pattern
D7 =	5’ resin bolts with a 3’x3’ regular pattern
DS =	3’ split-set bolts with a 4’x4’ regular pattern
"G"	When screen is needed
"B"	When metallic bands are needed

Table 16.12 shows standards for the different underground excavations in the mine.

Table 16.12 – Standards for the excavations in the mine

Excavation Type	Standard
Crosscut	D3G
Drilling Bay	D3G with screen 5' from the floor
Haulage Drift	D3G
Draw Point	D2B
Sill	D1
Sub-Level with Scraper	D1G

Excavation Type		Standard
Inclined Sub-Level		D3G
Raise		DS
Shrinkage Stope Span < 1.8m	Back	D2
	Hanging Wall	D3 with 3' bolts instead of 5'
	Footwall	D3 if span > 2.4m
Shrinkage Stope Span 1.8m to 5m	Back	D2
	Hanging Wall	D3
	Footwall	D3 if span > 2.4m
Shrinkage Stope Span > 5m	Back	D2
	Hanging Wall	D3 with 7' bolts instead of 5'
	Footwall	D3 if span > 2.4m
Room and Pillar Height < 2.4m		D5
Room and Pillar Height > 2.4m		D5 with 7' bolts instead of 5'

16.4 Hydrogeology

Figure 16.4, shows the various drainage basins in the Sleeping Giant mine area and the resulting flow direction of surface water.

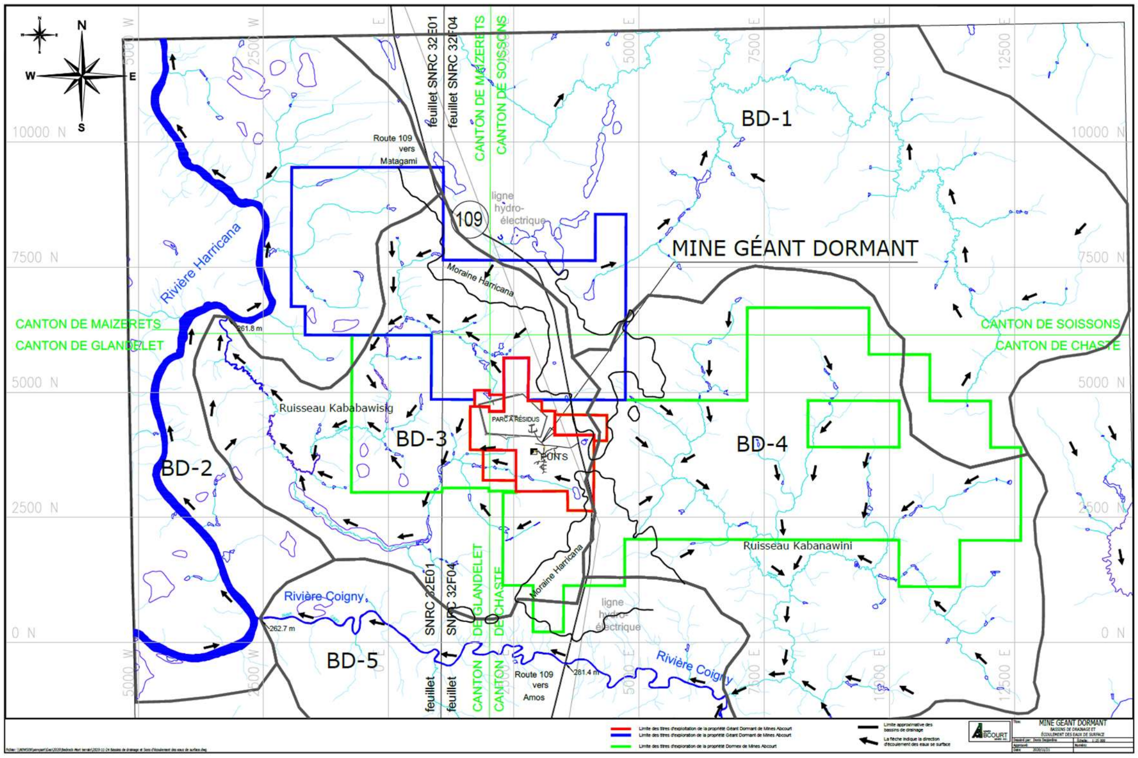


Figure 16.4 – Drainage basins in the Sleeping Giant mine area

Noteworthy features of Figure 16.4 include:

- The hydrographic network of the area of interest, with lakes and major rivers and streams (Rivière Harricana, Rivière Coigny, Ruisseau Kababawisig and Ruisseau Kabanawini).
- The boundary of the Sleeping Giant and Dormex properties, corresponding to all of Abcourt's mining and exploration claims in the Sleeping Giant Mine area. The position of Shaft No. 2, the tailings and the underground projection of the 235 level at surface were positioned for reference.
- The boundaries of Maizerets, Soissons, Glandelet and Chaste townships were superimposed for reference, as well as the position of Route 109, which links the towns of Amos and Matagami, and the hydroelectric line.

The study area includes parts of five (5) drainage basins (BD-1 to BD-05). Parts of BD-3 and BD-4 cover most of the Sleeping Giant and Dormex properties. BD-03 represents the Kababawisig Creek drainage and covers an area of approximately 4,000 hectares. It includes all surface mining infrastructure, including the shaft and tailings facility. BD-04 is the Kabanawini Creek basin and covers an area of approximately 6,100 hectares. The boundary between the Kababawisig and Kabanawini creeks basins corresponds to the roughly NS-trending Harricana moraine, on which Highway 109 rests.

Kabanawini Creek flows south to join the Coigny River, which flows 10 km further west into the Harricana River at a geodetic elevation of 262.7 m. Kababawisig Creek flows predominantly westward into the Harricana River at a geodetic elevation of 261.8 m.

A map of the topography of the bedrock was generated to assess the potential groundwater flow above it (Figure 16.5). As bedrock is predominantly impermeable material, knowledge of its topography helps determine the most likely points for groundwater resurgence.

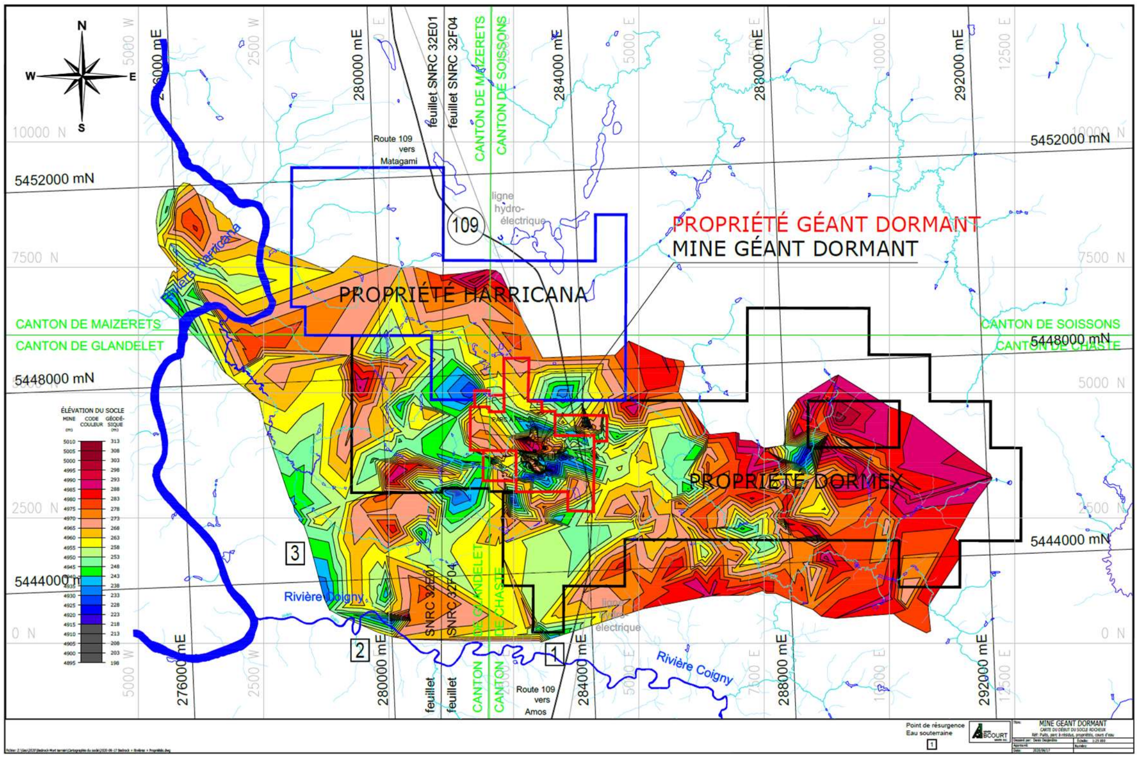


Figure 16.5 – Topographic map of the bedrock

A 3D model showing the projected bedrock surface was developed by triangulating the 3D points corresponding to the boundary between the overburden and the start of the bedrock from 818 boreholes (423 surface diamond-drill holes and 395 RC holes). The bedrock surface topographic map was generated by cutting the 3D surface at constant intervals (every 5 m of elevation).

The intervals between each level line have been coloured to allow rapid visual assessment of the underlying bedrock topography.

The bedrock topography map shows high topographic relief in the northern and eastern parts of the map. The mine area lies mainly within a topographic depression of about 5 km lateral extent, with a few mounds, all centred on the mine. Based on current knowledge, the most likely points of groundwater resurgence are in the southwestern part of the map at points 1, 2 and 3 on Figure 16.5.

Bedrock is not exposed in the Sleeping Giant mine area (Sleeping Giant and Dormex properties). Several historic RC drilling programs totalling 395 holes were carried out to document glacial dispersal trends of gold grains in Quaternary sediments (mainly in till), with the goal of generating and discovering potential gold targets and/or economic gold structures.

The RC drilling programs also documented the stratigraphy of the various Quaternary formations, including the position and thickness of horizons of clay, silt, sand, and till, as well as the depth-to and composition of the rock types directly underlying the Quaternary cover.

Of the 395 RC holes, 393 holes intersected the Objibway II Quaternary geological formation. This formation is abundant in the study area. It lies at the surface immediately beneath the first layers of decomposing organic matter. The uppermost part of this formation consists of layers of silty varved clays with a vertical thickness of up to 20.4 m. These sediments correspond to a very low energy glaciolacustrine environment and are generally distal. They are considered the most impermeable horizons.

A 3D surface (Figure 16.6) showing the thickness of the silty varved clay horizons of the Objibway II formation was modeled in the western part of the study area using data from the RC boreholes.

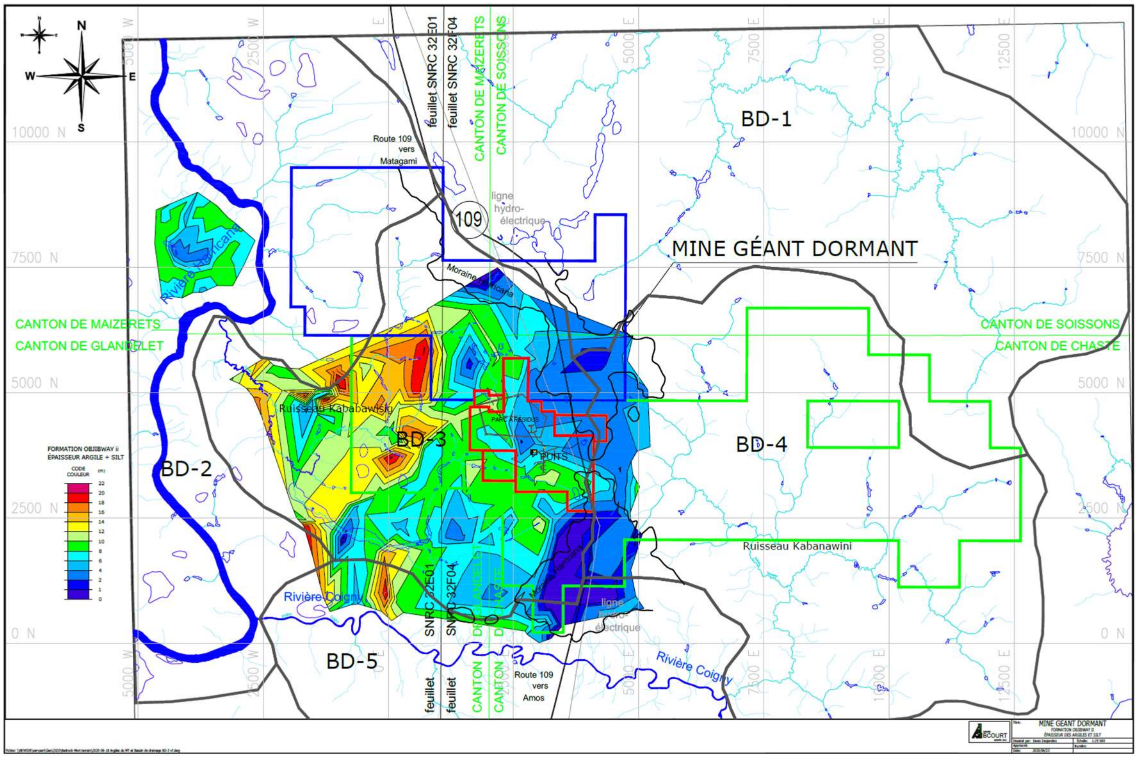


Figure 16.6 – Thickness of the silty varved clay layers of the Objibway II formation

The topographic map of the silty varved clay thickness was generated by cutting the 3D surface at constant intervals (every 2 m of elevation). The intervals between each level line corresponding to the thickness of the silty varved clays were coloured to facilitate rapid visual assessment of the clay thicknesses.

The thickest silty varved clays are found in the western sector, with general thicknesses of 10 to 20 m. The thinnest silty varved clays are found beneath the Harricana moraine. Thicknesses of silty varved clay in the immediate mine area generally range from 2 to 10 m. Levels of silty varved clay correspond to the most impermeable stratigraphic horizons in the overburden. They form an effective natural barrier between surface water and groundwater.

16.5 Mine Design

Sleeping Giant is designed as a mine with conventional track drift access that will use LH, R&P, and shrinkage techniques to exploit the underground mineralized material. The location, size, shape, orientation (dip), and physical properties of a mineral deposit generally determine the selection of the appropriate mining method.

The new project is built around the historic Sleeping Giant mine. Mine dewatering, waste management and pillar evaluation are all aspects that needed to be considered while designing the new mine around the old workings.

Level spacing is set at 45-60 m, based on existing levels, with sublevel spaced at an average of 15 m, where required. Including planned dilution, the minimal stope width is: 2.0 m for LH, 2.1 m for R&P, and 1.75 m for shrinkage. The production is divided by mining methods: LH, R&P, and shrinkage; respectively 25%, 32% and 43%. Drilling will be conducted with a combination of LH drills and jacklegs, depending on the mining method. Slushers may also be used to move material in captive stopes or sublevels.

Each level includes a lunchroom, a station and a maintenance bay located near the shaft station. Each level also has an explosives storage and a small capacity detonator magazine for short-term storage. The main explosives storage is on surface.

Currently, level 1175 does not have a lunchroom, maintenance bay or explosives storage.

Trackdrifts are excavated using portable pneumatic drills, LM loaders, electric locomotives and 4 tons minecars. Wheels loaders of type Cavo 320 are used to muck draw points and production stopes.

Fresh air will be supplied to the mine through the existing ventilation network.

The mining cycle for the LH method includes conventional half-timber raise development, sublevels development, sublevels cable installation, drop raise opening, production drilling and LH blasting.

R&P methods will allow extraction of shallowly dipping mineralized zones using development methods and leaving pillars in place to ensure rock stability. Considering the steep average dip, blasted muck may be left in place to facilitate worker movement in the stope. Slushers equipment are then used to empty the stope.

Shrinkage is used in stopes not suitable for LH or R&P methods and will allow for narrower stopes. Only the swell material is mucked at first, leaving most of the broken mineralized material (60%) in the stope to support the walls. The remaining mineralized material is then mucked and hauled by locomotive to the nearest ore pass loading door.

Whichever the method, confirmation of the stope completion by operations will initiate the engineering process, which includes geology, surveying, and mineralized material extraction validation.

Figure 16.7 presents an isometric overview of the project at completion. Figure 16.8 presents a longitudinal view facing north with the current level names.

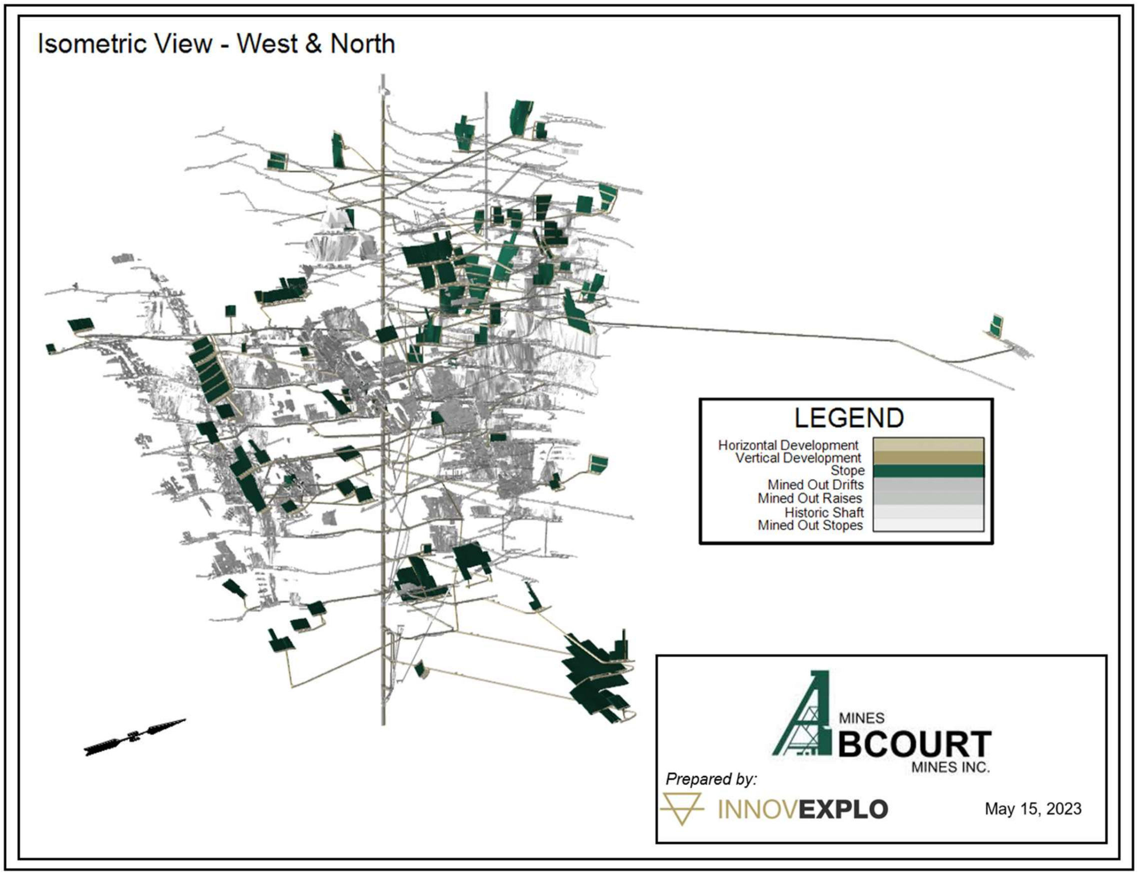


Figure 16.7 – Isometric view of the Sleeping Giant mine workings including the mine stopes optimization results

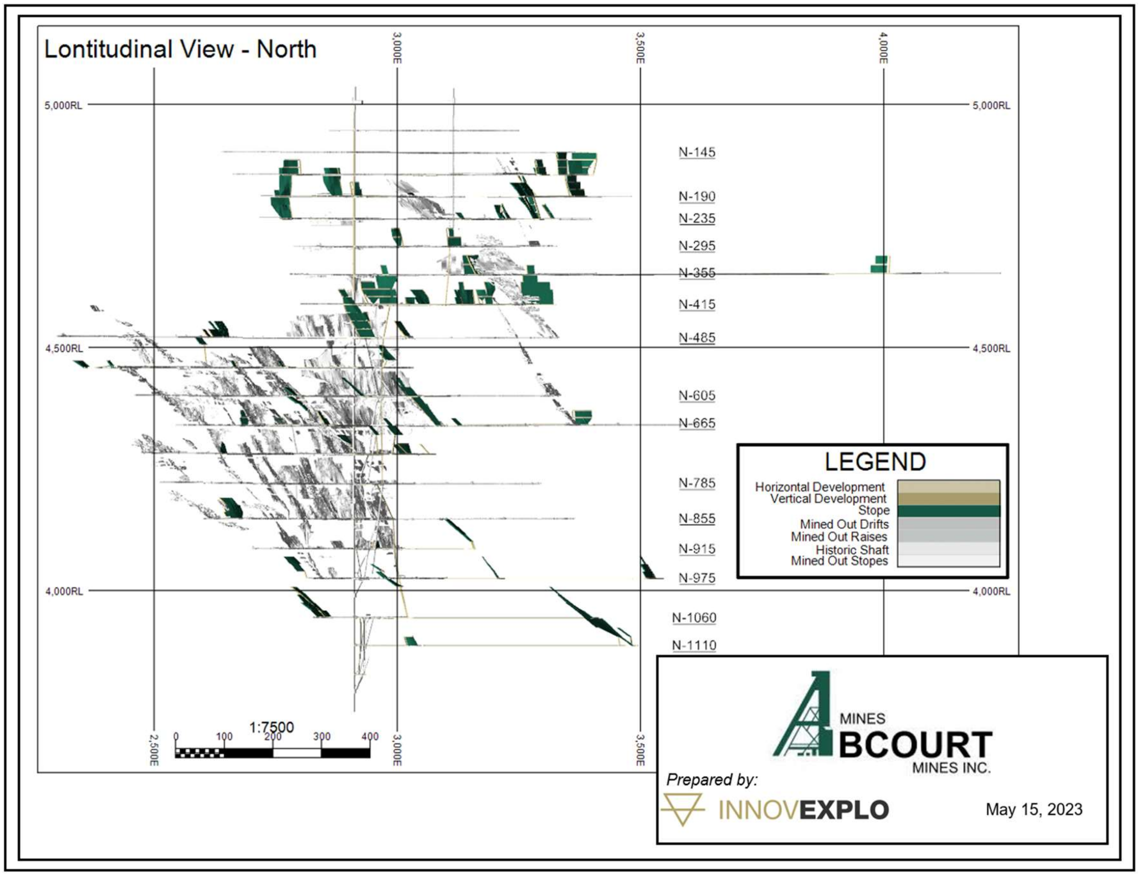


Figure 16.8 – Longitudinal view of the Sleeping Giant mine workings including the mine stopes optimization results

16.6 Stope Design

The Deswik Stope Optimizer™ (DSO) module was used on the mineral resource block model to generate mineable shapes that were subsequently used in the proposed design. Once the preliminary stopes were generated, a verification was made to remove any outlying stopes that would not be economic if development and mining costs were considered. Following the optimization, all stopes were reviewed to ensure the economic viability of the project.

Default dip and azimuth are inconsistent throughout the mineralized material. To refine the optimization process as much as possible, the optimization process has been divided into six (6) orientation groups. Based on vein evaluation from geological study, the six (6) average azimuth groups used are 15°, 45°, 73°, 104°, 140° and 166°.

Cut-off grade has been calculated considering the ELOS dilution included in the DSO.

Parameters used in the DSO module are presented in Table 16.13. Additional key design parameters are presented in Table 16.14.

Table 16.13 – DSO parameters for underground mining

Parameters	LH	R&P	Shrinkage
Density (t/m³)	2.85	2.85	2.85
Optimization field	Au	Au	Au
COG diluted (g/t)	4.50	5.25	5.05
Default dip (°)	Var.	Var.	Var.
Maximum dip (°)	90	43	90
Minimum dip (°)		0	43
Strike azimuth	Var.	Var.	Var.
Maximum strike change (°)	60	60	60
Sub-shapes	No	No	No
Optimization length (m)	5	5	5
Minimum mining width without dilution (m)	1.7	1.7	1.7
Minimum mining width with dilution (m)	2.0	2.1	1.75
HW dilution (m)	0.2	0.3	0.025
FW dilution (m)	0.1	0.1	0.025
Stope maximum side-length ratio	2.5	1.5	2.5

Notes:

- Cut-Off grade (COG): LH 4.50 g/t Au, R&P 5.25 g/t Au and Shrinkage 5.05 g/t Au
- Au price US$1,650 per troy ounce
- The exchange rate used was 1:1.30 US$/C$.

Table 16.14 – Key design parameters

Parameters	Units	LH	R&P	Shrinkage
COG diluted (Au)	g/t (Au)	5.25	6.50	5.20
Minimum mining width	m	2.0	2.1	1.75
Mining height	m	15	15	15
Mining length	m	5	5	5
Minimum HW angle	°	52	0	43
Minimum FW angle	°	52	0	43
Mining recovery	%	95	81	95

Notes:

- Cut-Off grade (COG): LH 5.25 g/t Au, R&P 6.50 g/t Au and Shrinkage 5.20 g/t Au
- Au price US$1,650 per troy ounce
- The exchange rate used was 1:1.30 US$/C$.
- Mining height variable, optimized on gradient lines

16.7 Main infrastructure

The shaft is approximately 1230 m deep. There are 18 production levels in total. The shaft has been studied several times. The upper levels are spaced vertically every 46 m, whereas the lower levels are spaced 60 m apart. Exceptions to this rule are levels 415 and 485, which are 70 m apart; levels 785 and 855, which are 70 m apart; and levels

975 and 1060, which are 85 m apart. Shaft stations are typically 6.8 m wide x 23 m long x 3.0 m high.

There are loading stations located below levels 415, 785, 975 and 1175. Loading station 415 is no longer in service. The loading station at 785 is only used for sterile material, whereas the loading station at 975 processes both mineralized material and waste rock (from ore and waste pass system).

A loading pocket is located below levels 1060 and 1115, allowing development excavations at these levels. There is a system of mineralized material and waste pass from level 055 to level 975. There is a discharge station for mineralized and sterile material with a grizzly at all levels. The dimensions of the ore passes are typically 2.1 m x 2.1 m.

Abcourt plans to complete the ore pass to serve levels 1060, 1115 and 1175. The waste rock from levels 1060 and 1115 will be deposited into their respective loading pockets to allow for further development.

Several additional pieces of critical infrastructure have already been installed for the underground mining operations (some assets may need rehabilitation and restoration before production starts), as follows:

- Electrical network and stations
- Pumping network and sumps
- Ventilation network and egress way (manways)
- Refuge (lunchroom)
- Explosive storage and cap magazine
- Mineralized material and waste dumps
- Workshop and battery charging stations

New infrastructures will be constructed where new developments necessitate them, notably in levels 1115 and 1060 (new levels). In addition to the components listed above, all surface support infrastructures necessary for the underground operations (e.g., truck shop, mine dry, mine offices), are already present. No major changes to surface infrastructures are projected with the new production.

16.7.1 Production shafts and hoisting system

The mine hoist is a double drum powered by two 518 hp synchronous motors. The headframe is made of steel and has three compartments. The headframe has a silo for mineralised material and a waste rock silo with a capacity of approximately 500 and 200 tonnes each. The shaft transporters consist of a cage/skip configuration in a compartment and a skip/counterweight in the other. The skips are made of aluminum with a capacity of 4 tons.

The production shaft ("Shaft #2") is a rectangular 2.74 m x 6.8 m shaft consisting of three (3) 1.8 x 1.8 m compartments. The wood support and guides are made of Douglas fir lumber vertically spaced 2.1 m. There is also a manway compartment. A water collector is located above each level. A fourth compartment below level 355 was used for previous shaft deepening operations. This fourth compartment is currently not used.

The services inside Shaft #2 are installed in the manway compartment and include a 203 mm (8”) diameter steel pump line, an air line 152 mm (6”) diameter steel tablet, steel industrial water line 102 mm (4”) diameter, 51 mm (2”) diameter HDPE drainage line and miscellaneous electrical and communication cables.

16.8 Rehabilitation

Rehabilitation will take place on all levels. Existing track drifts will be rehabilitated to allow production on most levels (except level 785 where only access to the emergency egress way is rehabilitated).

Extensive ground condition evaluation has been on-going. For the purposes of planning, rehabilitation has been categorized into three (3) types. All existing infrastructures have been categorized as medium rehabilitation. Table 16.15 shows the proportion of material used for each rehabilitation type and Table 16.16 shows the performance and distribution for each type.

Table 16.15 – Rehabilitation material utilisation

Rehabilitation type	Rock bolts	Screens	Pipes
Rehab – Full with Support	100%	25%	10%
Rehab – Full	100%	10%	10%
Rehab – Medium	100%	5%	10%

Table 16.16 – Rehabilitation performance and proportion

Rehabilitation type	Performance (m/day)	Eq. m	Proportion
Rehab – Full with Support	16.0	9.0	22%
Rehab – Full	20.0	3.1	63%
Rehab – Medium	26.0	2.1	15%

16.9 Mine Design Criteria

Track drifts are designed 2.7 m wide by 2.7 m high, whereas draw points and sublevels are 2.7 m wide by 2.4 m high and 2.5 m wide by 2.4 m high, respectively.

Conventional half-timber raises (2.4 m by 2.1 m) are used in LH and R&P stopes, wherever equipment is required in a sublevel. Conventional timber raise is used as a service raise, mostly in shrinkage stopes. If slushing distance is too long, an additional conventional raise is developed and used for mineralized material only. Various development parameters are summarized in Table 16.17. Gradient shown in the table are indicative only and not represented in the 3D model.

Table 16.17 – Mine Design Parameters

Development Heading	Width (m)	Height (m)	Gradient
Track drift	2.7	2.7	0.5%
Draw points	2.7	2.4	0.5%
Sublevels	2.5	2.4	0.0%
Development Heading	**Width (m)**	**Length (m)**	**Gradient**
Conventional raise	1.8	1.8	N/A
Conventional timber raise	1.8	2.1	N/A
Conventional half-timber raise	2.4	2.1	N/A

16.10 Emergency Egress

The existing ventilation network will be used as the main emergency egress way. The existing raises will be rehabilitated, as needed. Only one additional raise is required to complete the egress/ventilation network and incorporate the new lower levels, at level 1060.

As per the legislation, additional raises have been added whenever a stope length is above 30.0 m and has only one egress. R&P, as per the legislation, do not require a second egress.

16.11 Mining Methods

Mine development will employ numerous production fronts to maximize productivity and flexibility to reach the 350 tpd target. Three main mining methods will be employed: LH, R&P, and shrinkage. See Table 16.13 for a list of the parameters applied to each method optimization process.

Table 16.18 presents the distribution of tonnage for each method and Figure 16.9 presents the mining methods in the 3D model.

Table 16.18 – Mining method distribution

Mining method	Tonnage	%	Ounces (Au)
Long Hole	197 200	27%	45 800
Room & Pillar	309 000	43%	74 900
Shrinkage	214 100	30%	66 900

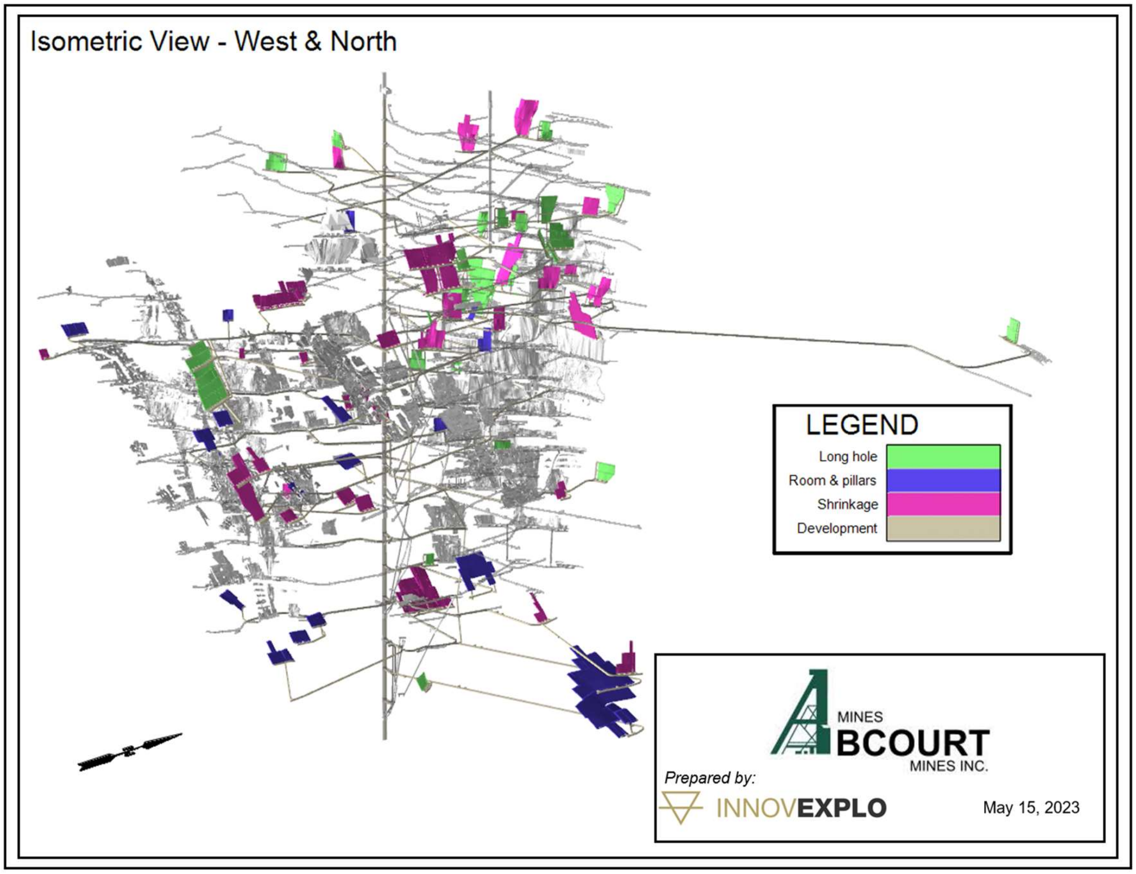


Figure 16.9 – Mining methods isometric view

16.11.1 Long hole (LH)

LH mining is the latest mining method to be applied at the Sleeping Giant mine. It has been used in the past, but only as test stopes, with varying results. This method allows for high productivity, with lesser manpower than the conventional methods.

A track drift is generally developed parallel to the veins to allow for material haulage. Draw points are then developed perpendicular to the vein, spaced at an average 15 m. A conventional half-timber raise is then developed vertically, normally on one extremity of the vein, outside the mineralized material (a 4 m pillar is targeted). Sublevels are then excavated with a maximum 15 m spacing, with opening cuts made on each sublevel to facilitate further development. Cables are installed and a drop raise is blasted for the primary opening. Individual 5 m panels are then mined using a longitudinal retreat method. Mineralized material is fully mucked between each blast from the draw points using Cavo equipment and is then hauled to the mineralized material dump.

Blasted stopes hold mineralised material until it is ready for mucking. Following mucking, the next panel is blasted into the opening. The material stored in the stopes also provides stability to the walls.

Figure 16.10 and Figure 16.11 present a section view and a plan view of the LH method.

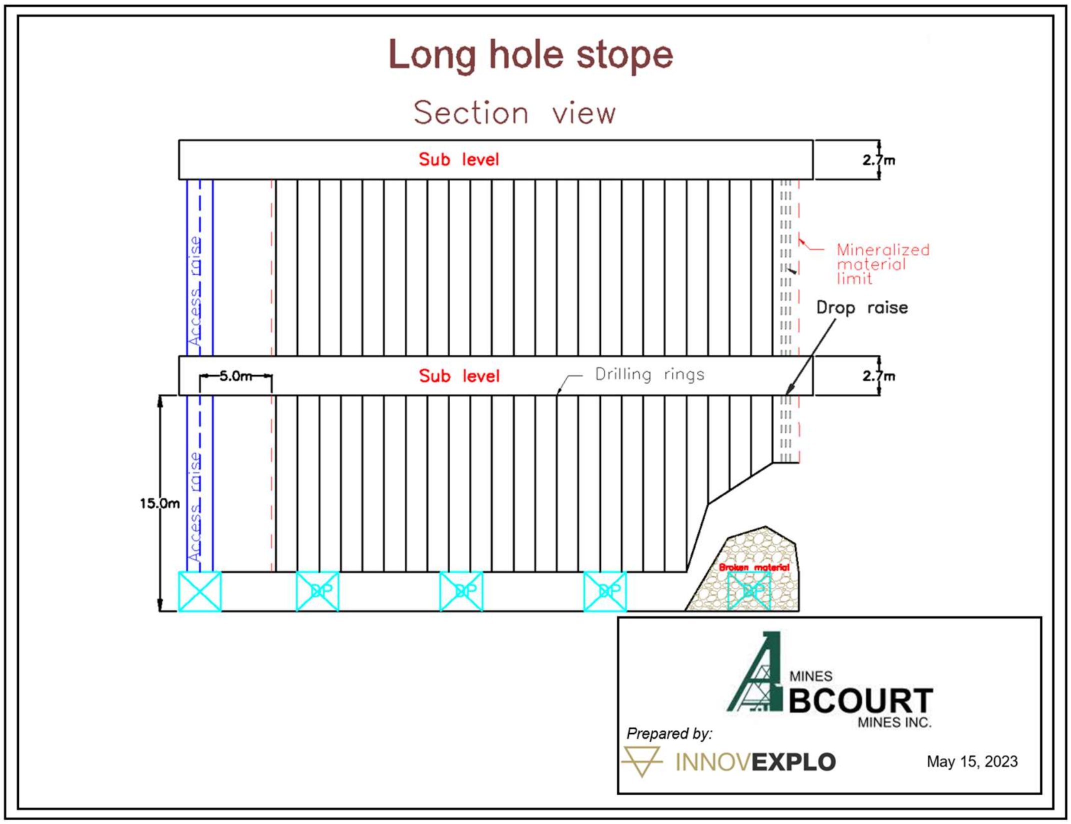


Figure 16.10 – LH section view

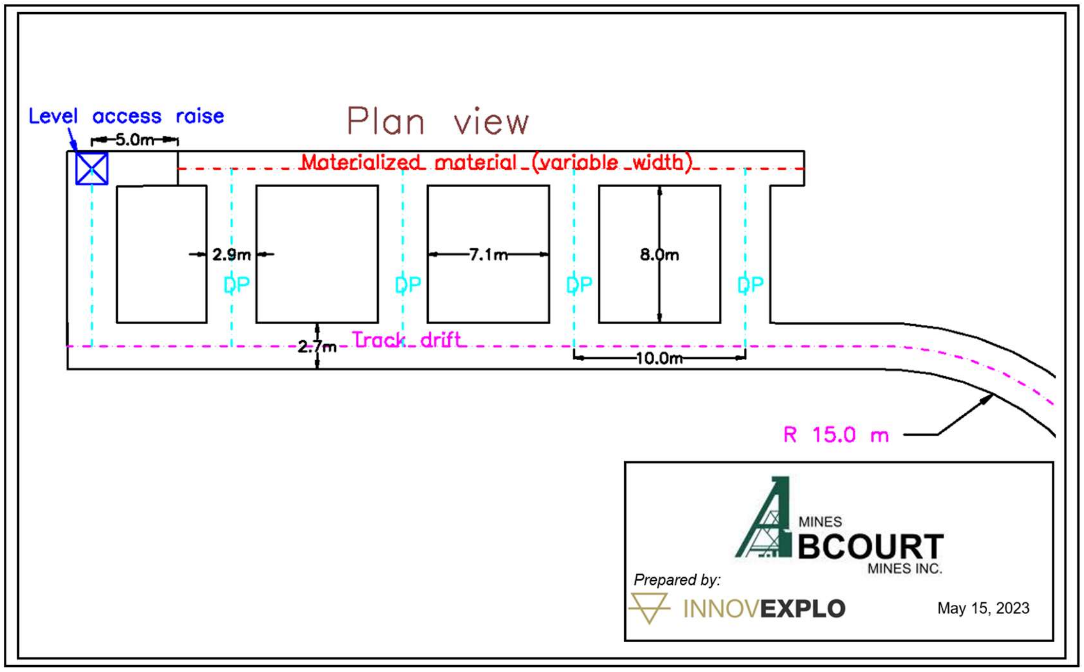


Figure 16.11 – LH plan view

Figure 16.12 shows one of the main LH mining horizons (levels 415 and 485) of the Project in 3D view and presents a typical on-going sequencing of the mining method.

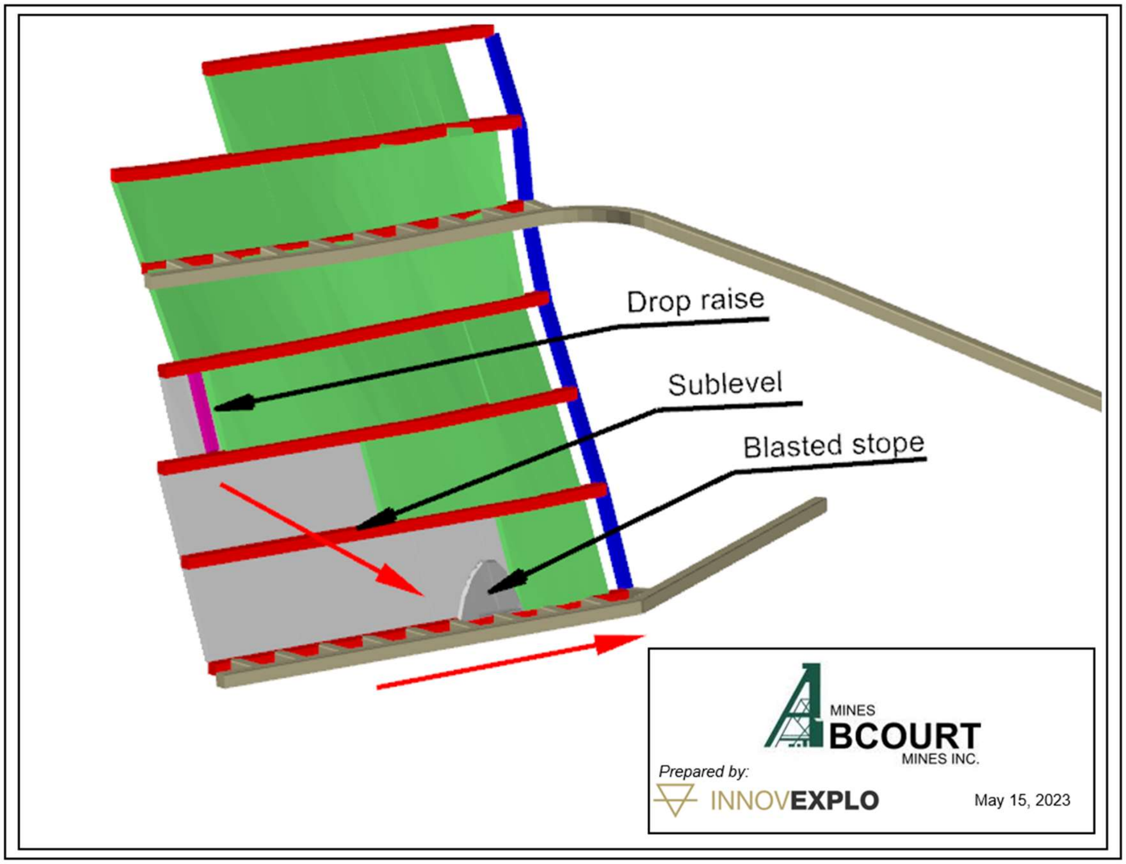


Figure 16.12 – LH mining horizon

16.11.2 Room & Pillar (R&P)

The R&P method is used for sub-horizontal (shallow-dip) mining and use square pillars to stabilize a large excavation, creating arrays of rooms and pillars. At Sleeping Giant the dip tends towards 43° and adds significant challenges to this method.

Primary mucking of the blasted rooms allows for 75% of the mineralized material to be recovered. The remaining material is left on the footwall to facilitate movement for the workers and is recovered when the stope is completed. Slushers are used to move the mineralized material toward the draw points or conventional raise (if the stope doesn't connect with the main level).

Draw points, designed to respect an 8.0 m pillar between the mineralized material and the track drift, are spaced up to 40.0 m. Planned pillars dimensions are set at 3.0 m by 3.0 m, whereas the maximum width between pillars is 6.0 m. Pillars may be recovered at the end of the stope, but this additional material is not considered in the current study.

Figure 16.13 presents a section view of the R&P method.

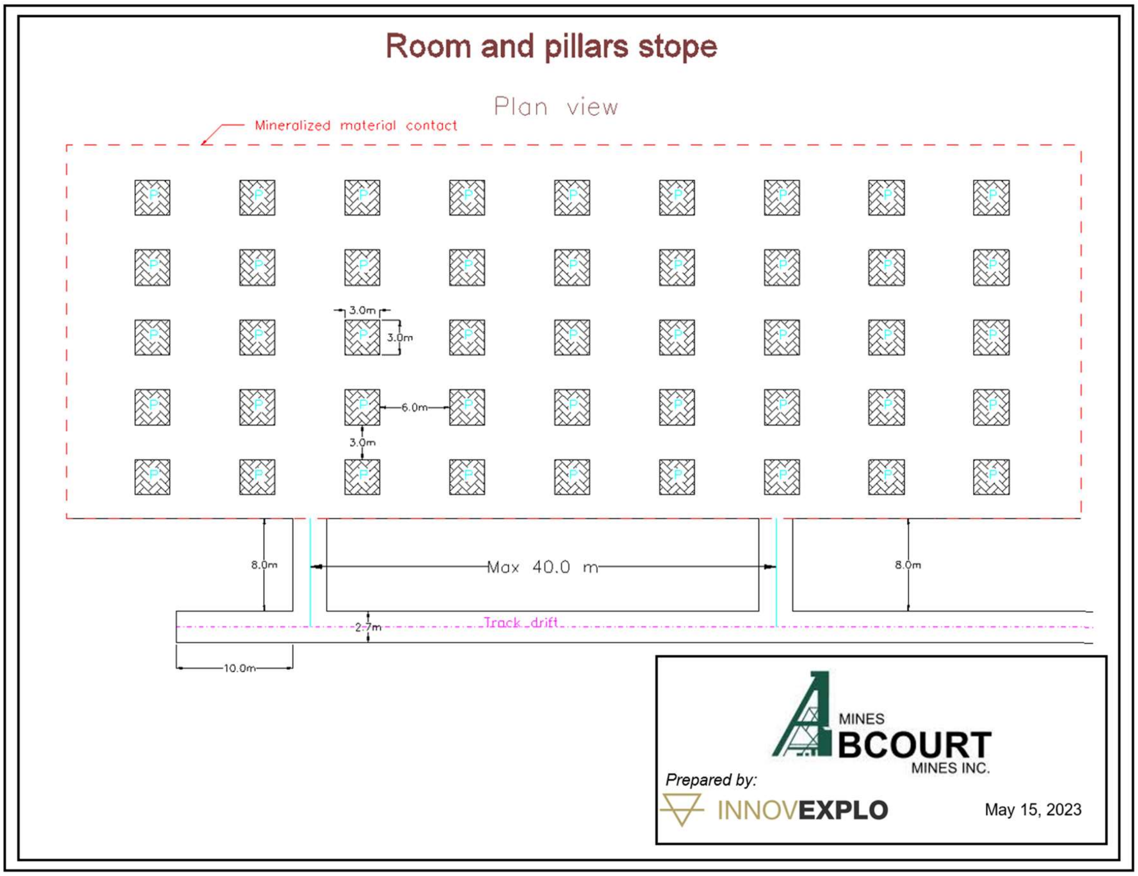


Figure 16.13 – R&P plan view

Figure 16.14 shows one of the main R&P mining horizons (levels 1060 and 1115) of the project in 3D view and presents the overview of the mining method at completion.

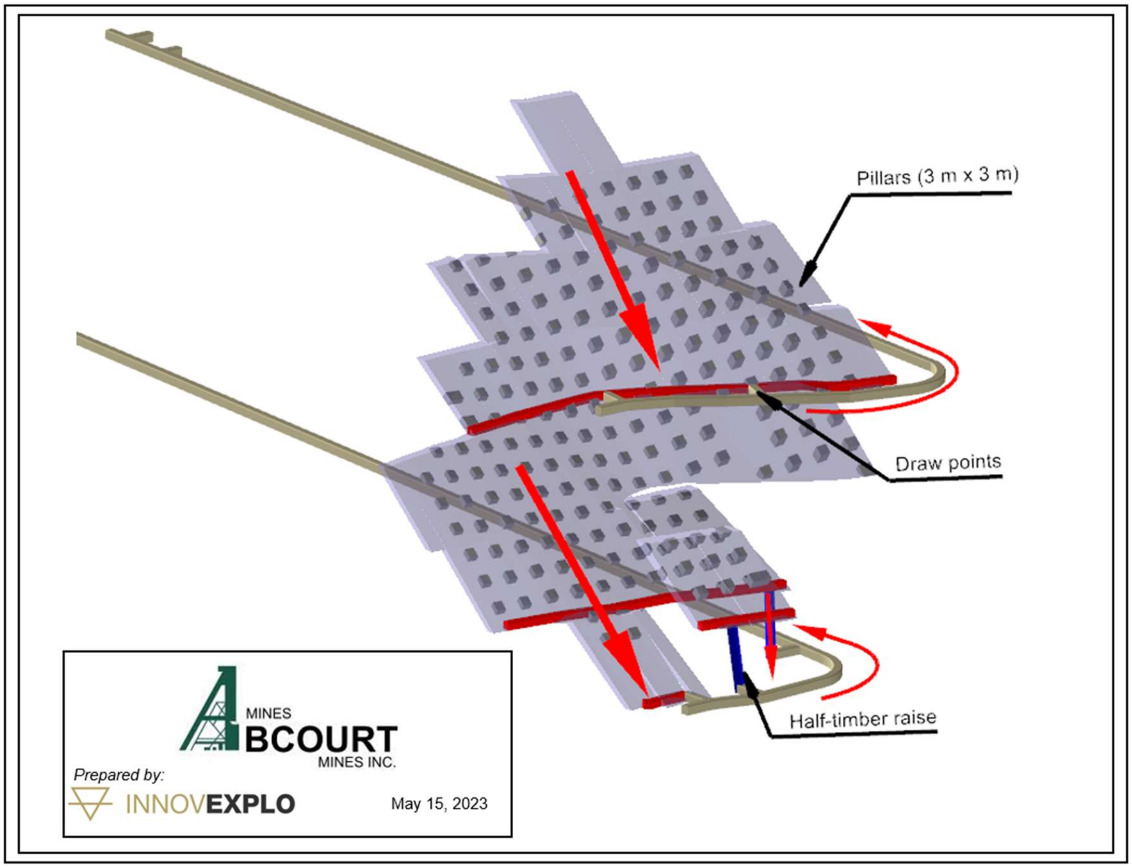


Figure 16.14 – R&P mining horizon

16.11.3 Shrinkage

Shrinkage stoping is used for steep and narrow mineralized material bodies, not suitable to LH mining. It relies on broken material being left in the stope to be used as both working floor and wall support. During mining, only the swell material is mucked at first (equivalent to 40%); the remaining material is mucked out at the end of the stope.

Draw point are also designed with an 8.0 m length and spaced every 7.1 m (10.0 m centreline to centreline). As stated, the stopes hold the mineralised material not mucked during the first mucking phase (swell mucking). The material stored in the stopes also provides stability to the walls, until ready for the final mucking phase.

Figure 16.15 and Figure 16.16 present a section view and a plan view of the shrinkage method.

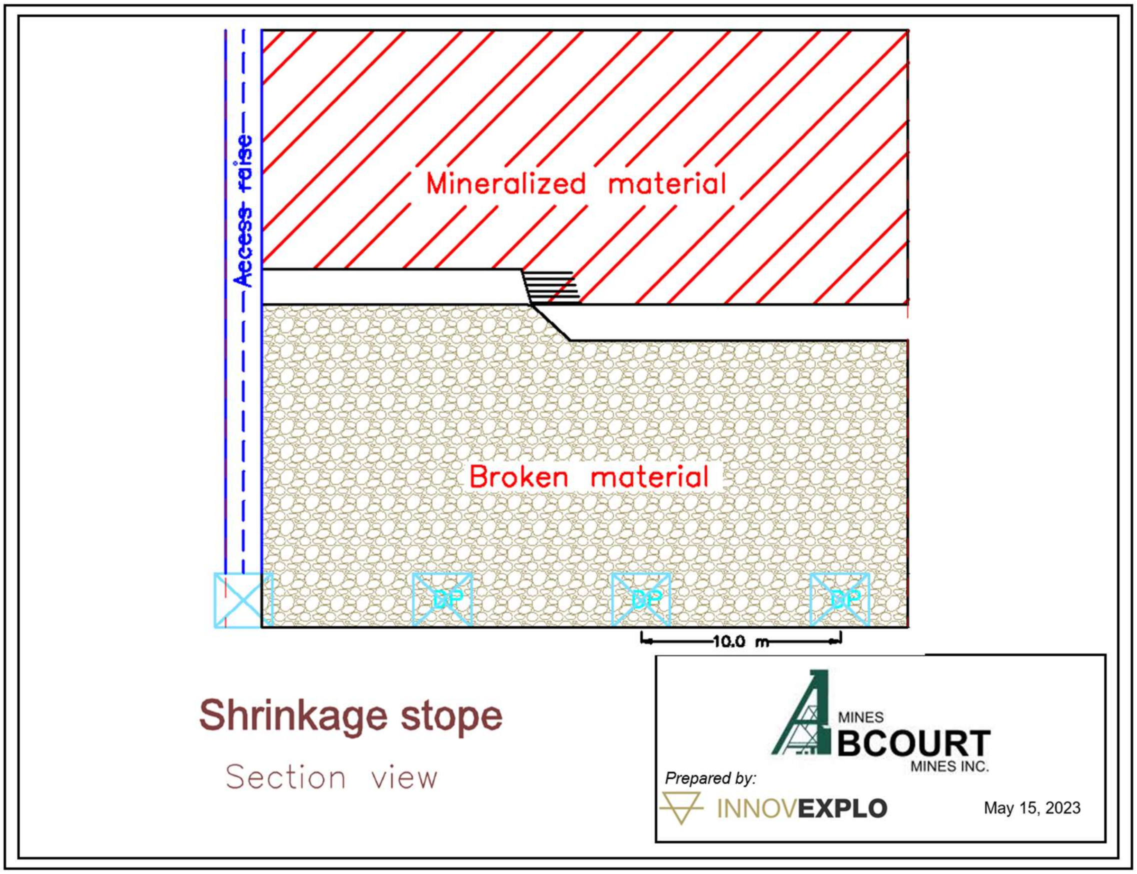


Figure 16.15 – Shrinkage section view

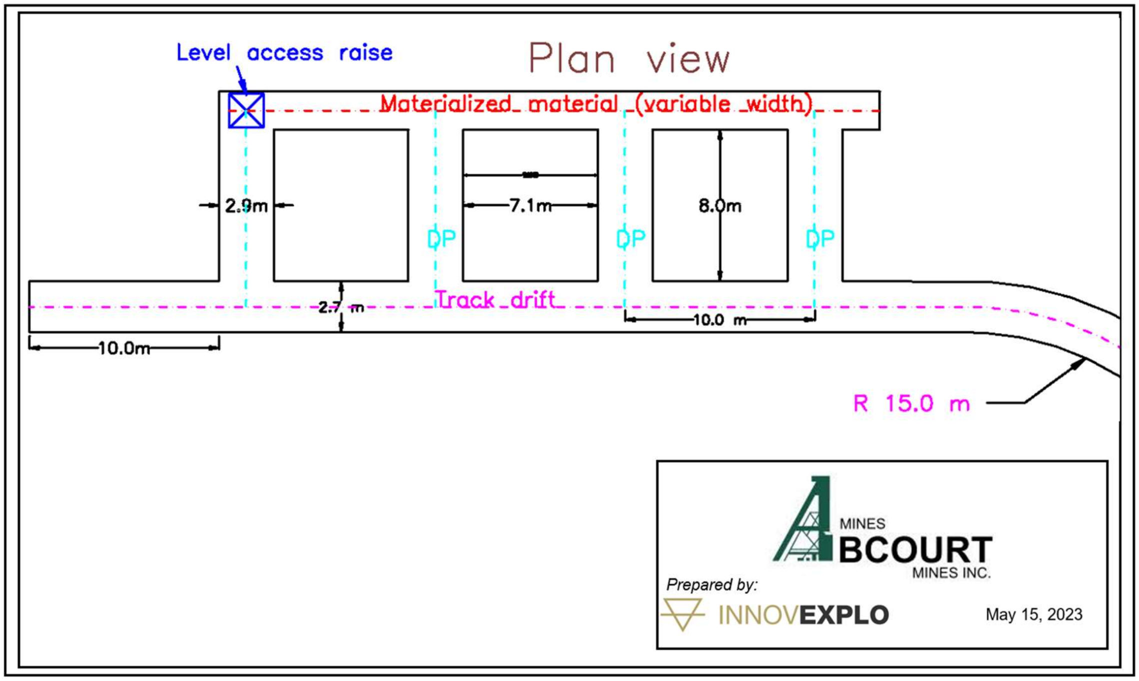


Figure 16.16 – Shrinkage plan view

Figure 16.17 shows one of the main shrinkage mining horizons (level 1060) of the project in 3D view and presents a typical on-going sequencing of the mining method.

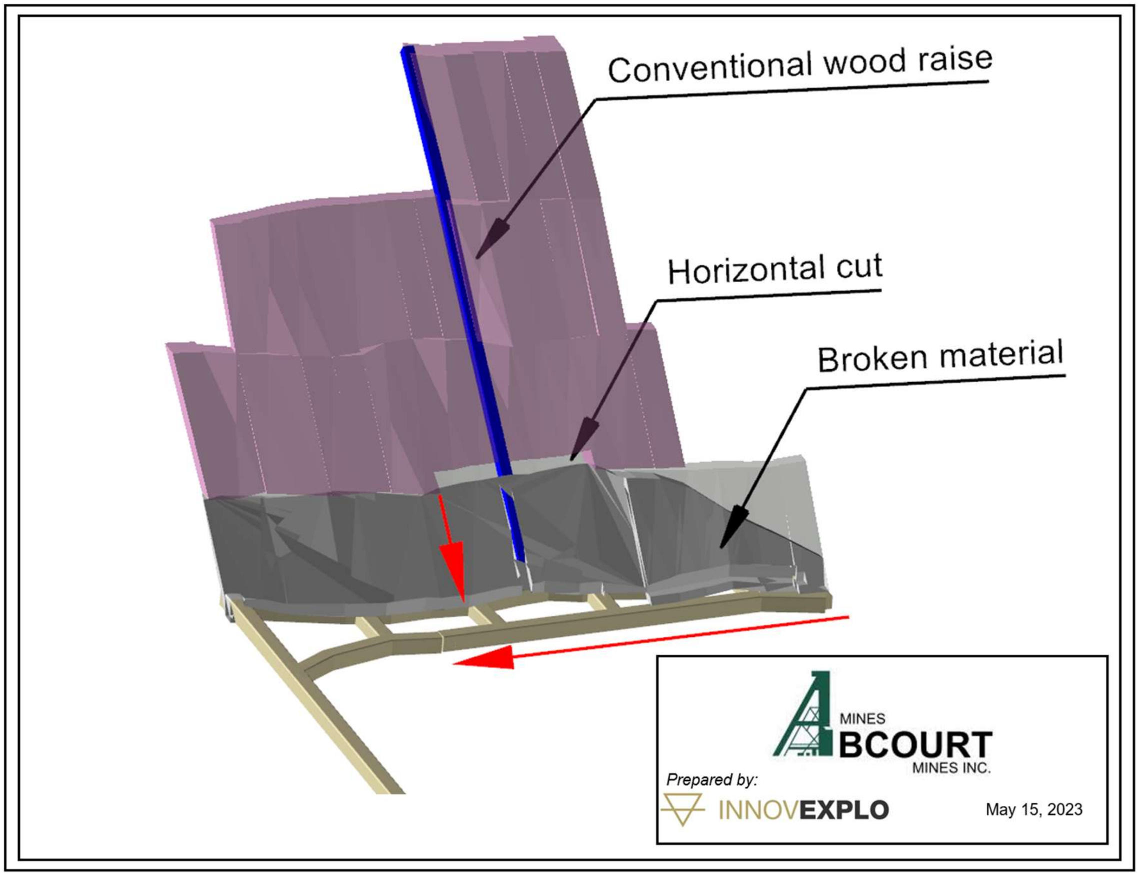


Figure 16.17 – Shrinkage mining horizon

16.12 Mine Services

16.12.1 Mine water pumping

There are three (3) main pumping stations in the mine, located around the Shaft #2 on levels 235, 415 and 975. Other pumping stations are located on levels 1060, 1115 and 1175. Muddy water above each of these levels is directed to the sumps using drain holes and gravity. The dewatering strategy includes seven lifting stages, starting from the shaft bottom. The mine water is pumped from one station to the next, up to the surface where it is treated.

Table 16.19 lists the underground pumping stations. The permanent mine pumping diagram is shown in Figure 16.18.

Table 16.19 – Underground pumping stations

Location	Pumping construction	Pumping equipment
Level N-235	Clear water sump and	2 x 125 HP Mather & Platt pumps
	Dirty water sump	1 x 125 HP Mather & Platt pump (Backup)
Level N-415	Clear water sump and	1 x 125 HP Mather & Platt pump
	Dirty water sump	1 x 125 HP Mather & Platt pump (Backup)
Level N-975	Clear water sump and	1 x 250 HP Technojet pump
	Dirty water sump	1 x 250 HP Technojet pump (Backup)
Level N-1060	Dirty water sump	1 x 19 HP Submersible pump installed in a metal tub
Level N-1115	Dirty water sump	1 x 19 HP Submersible pump installed in a metal tub
Level N-1175	Dirty water sump	1 x 19 HP Submersible pump installed in a metal tub
Shaft Bottom	Dirty water sump	1 x 19 HP Submersible pump installed in a metal tub

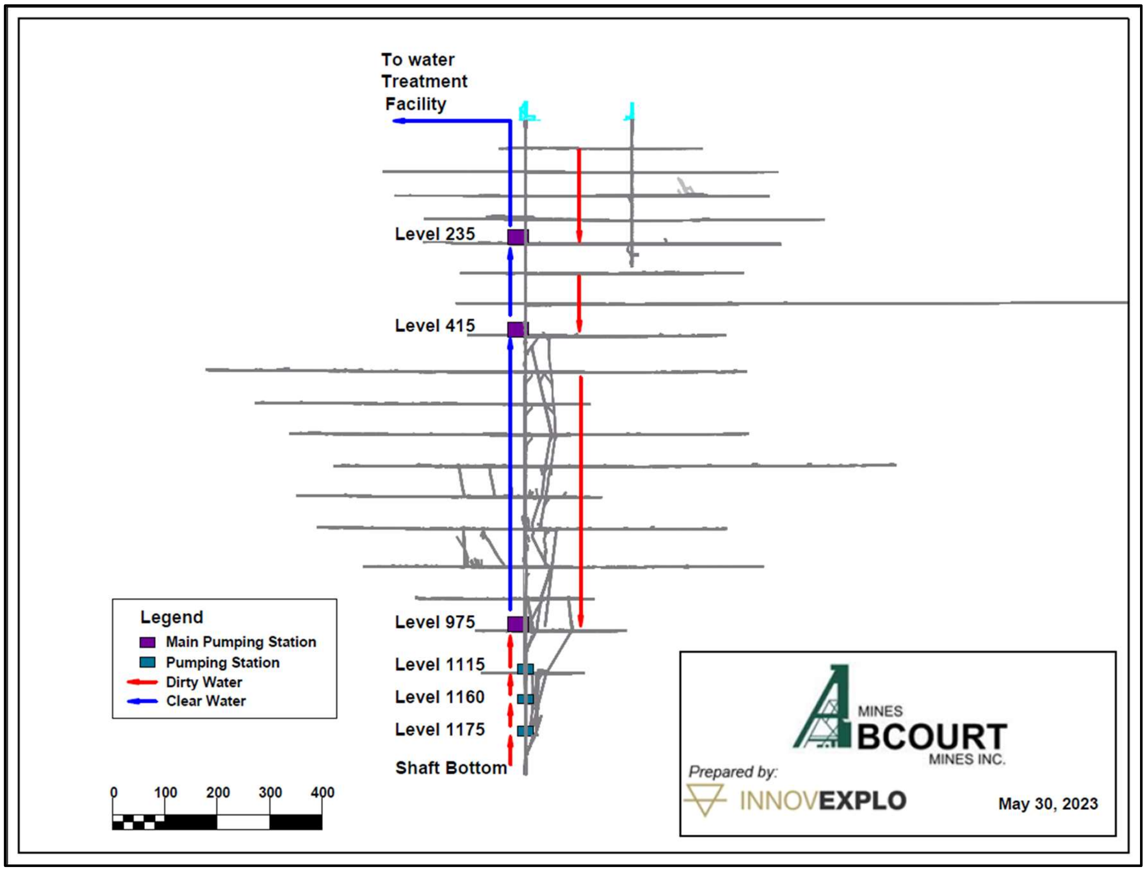


Figure 16.18 – Mine pumping diagram

16.12.2 Electrical and communication network

Electrical energy is distributed underground to the main power stations at 2400 volts. This distribution has a total capacity of 400 amps. The power distribution on the levels is at 600 volts. Figure 16.19 shows a diagram of the 2400 volts distribution to the main underground stations.

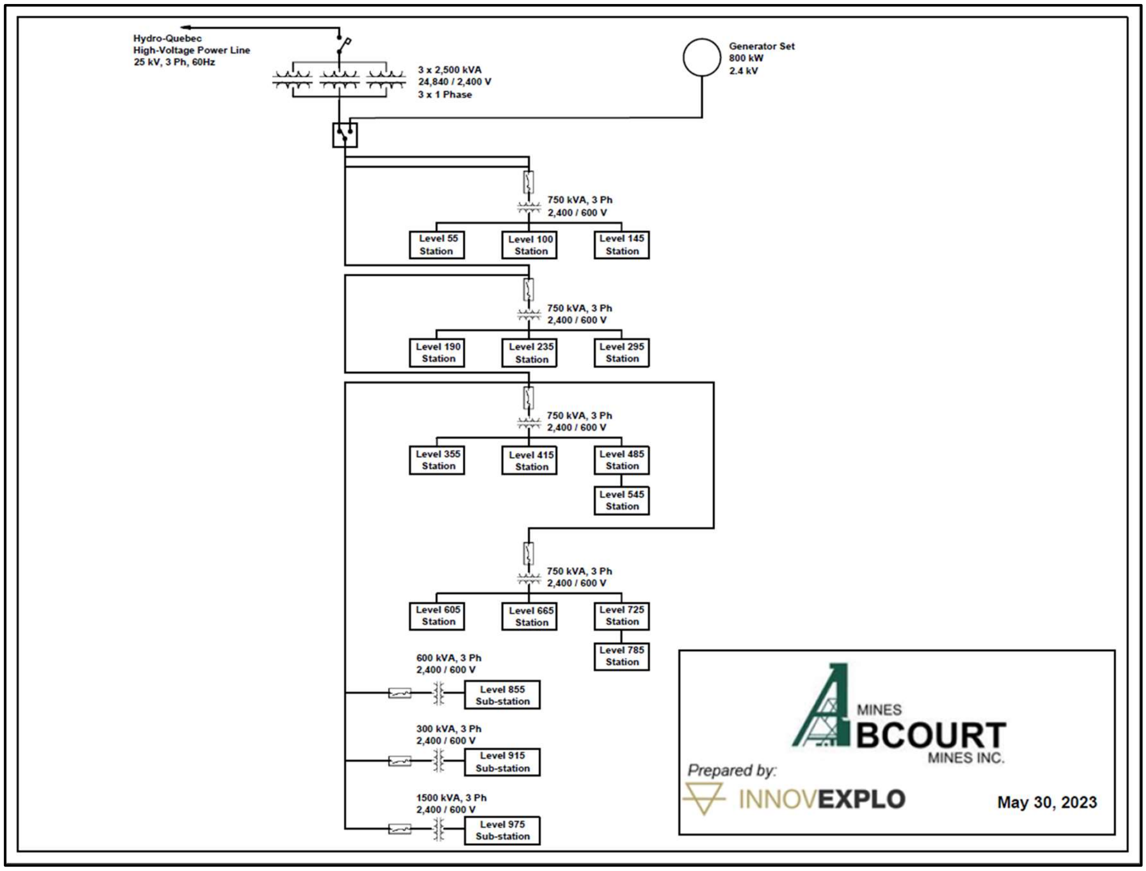


Figure 16.19 – Simplified diagram of the 2400/600V underground power supply

There are two underground communication systems at Sleeping Giant. One is a mine wired telephone system that provides voice communication between the mine offices, winch room, shaft building, shaft stations, dining rooms and other designated key locations. The second communication system is a radio system using a network of underground radiant cables and surface antennas. It is used for voice communication between almost all underground and surface areas. It cannot be used for the transmission of data such as fan condition, air quality, pump condition and sumps water level.

16.12.3 Ventilation

Two main fans and a surface heating system are installed on the former Shaft #1 and provide fresh air to the mine. Air is pushed into Shaft #1 down to level 235, then distributed further down through raises and stopes. Contaminated air rises to surface through Shaft #2. The main surface fans are two (2) 42-inch-diameter axial fans, with 26-

inch hub, 100 hp and 1770 rpm. Each fan has a capacity of 50,000 cfm. The total available airflow is 100,000 cfm. Fresh air is distributed through the emergency exits, active and old stope using auxiliary ventilation. The intake (Shaft #1) is provided with a manway serving as escapeway from level 235 to surface.

Air flow on the levels is controlled using ventilation doors and control walls. The air is pushed towards the faces of development and active production sites through fan pipes and auxiliary fans. The mine air heating plant is a direct propane gas heating system with a capacity of 11 MMBtu. The propane tank has a capacity of approximately 113,000 litres (30,000 US gallons) sufficient to last several weeks in the coldest winter months. The need for fresh air is based on Quebec mining regulations specifying a minimum 15 m^3/s (530 cfm) per person and on the number of underground diesel equipment. No diesel equipment is currently planned to be used underground. The total need for fresh air is estimated at less than 50,000 cfm. Figure 16.20 illustrates the primary ventilation routes.

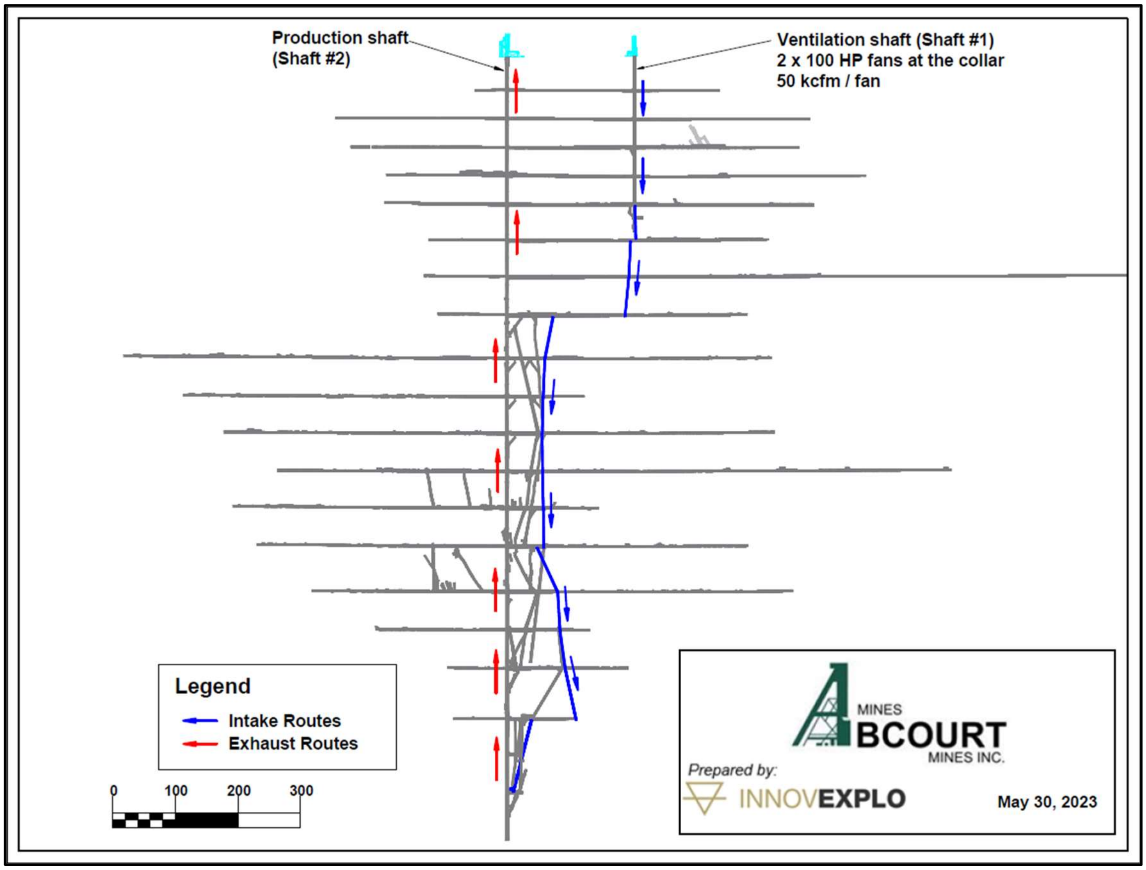


Figure 16.20 – Primary ventilation network

16.13 Underground Production Plan

Underground operations will be carried out on a continuous schedule of 7 working days and 7 days off, day and night shifts, with 10 hours shift. The planned development and stope production in mineralized zones will start after rehabilitation of the ventilation raises, serving as primary ventilation routes and/or secondary egress, connecting underground workings to the surface. The manway serving as escapeway from level 235 to surface has already been rehabilitated, whereas from level 295 to level 1060, all the escapeways need full rehabilitation. On the levels where production stopes are planned, the development is carried out after rehabilitation of the existing track drift, starting from the production shaft station, with possible track drift extension toward the mineralized zone, where needed. For all levels below level 295, stope production and related development work will start only when the ventilation fresh air raise provided with wood manway serving as level second egress is completed.

Development of mineralized material is projected to begin in the second quarter (Q2) following the start-up of the project with one development team. Stope production will start in early Q3 on level 1060 where a shrinkage stope and a R&P stope are to be mined. Production on level 1115 would begin in then fifth quarter (Q5) with R&P mining. During the first year of operation, the total projected production is 3.5 k tonnes of mineralized material grading 7.25 g/t Au sent to the mill, and 81.8 k tonnes grading 7.12 g/t stockpiled.

The mine is projected to be running at full production after 15 months when 350 tpd is reached. Two (2) development teams will be required to meet the production plan. During the second year of production, a total of 110.7 k tonnes of mineralized material will be hoisted with an average grade of 6.76 g/t, extracted from various stopes on levels 145, 355, 415, 485, 725, 1060 and 1115, including two (2) R&P stopes, six (6) shrinkage stopes, and two (2) LH stopes. A quantity of 29.3 k tonnes @ 7.25 g/t remains stacked on the stockpile area, while 127.7 k tonnes @ 6.82 g/t is sent to the mill.

From year 2 to year 6, mineralized material is no more stockpiled, and all the tonnes mined are sent to the mill. Apart levels 055, 100, 785 and 1175, all the other levels will be in production.

A total of 720,000 tonnes of mineralized material is planned to be mucked out of the mine by August 2030, on a seven (7) years production plan. Approximately 43 % of the mineralized material will come from shrinkage stopes. R&P stopes and LH stopes will account respectively for 30 % and 27 % of the production plan.

Abcourt plans to continue operating the mine as more resources will be defined following an on-going exploration program, or an update study following an increase in the price of gold.

The development mine plan is summarized in Table 16.20 and the production mine plan in Table 16.21.

Table 16.20 – Mine plan - Development

Development	Unit	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Total
Rehabilitation	m	7,460	930	2,060	780	2,600	2,260	0	16,090
Horizontal development	m	3,370	3,290	2,560	2,100	1,930	1,950	250	15,440
Vertical development	m	570	620	530	490	540	440	180	3,360
Waste Produced	t	67,700	54,910	55,010	34,490	37,670	37,270	4,790	291,860

Table 16.21 – Mine plan - Production

Production	Unit	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Total
Tonnes Mined	kt	20.5	110.7	123.6	127.5	126.2	127.5	84.2	**720**
Gold Grade	g/t	7.25	6.76	8.37	9.63	7.31	7.68	9.20	**8.10**
Gold (mined)	koz	4.8	24.0	33.2	39.5	29.7	31.5	24.9	**188**
Stockpile Inventory	kt	81.8	29.3	-	-	-	-	-	**111**
Gold Grade	g/t	7.12	7.25	-	-	-	-	-	**7.15**
Gold (mined)	koz	18.7	6.8	-	-	-	-	-	**26**
Tonnes Milled	kt	3.5	127.7	123.6	127.5	126.2	127.5	84.2	**720**
Mineralized material per day (average)	t	10	360	348	359	356	359	237	**336**
Gold Grade	g/t	7.25	6.82	8.37	9.63	7.31	7.68	9.20	**8.10**
Gold (milled)	koz	0.8	28.0	33.2	39.5	29.7	31.5	24.9	**188**

16.13.1 Production performance

Detailed cycle times were calculated for the main underground activities to maintain an accurate mining scheduling. The operational parameters used are detailed in Table 16.22. Each day includes two 10-hour shifts and considers all related operational activities such as shift changes, lunch break, refuelling, loss of time and transportation to workplaces. There are two (2) production teams per shift and two (2) miners per team.

Table 16.22 – Operating Parameters

Operating Parameters	Units	Quantity
Average production days per year	days	355
Number of shifts per day	shifts	2
Hours per shift schedule	hours	10

Mine operations are scheduled on a 355 working days per year schedule. Evaluation of projected production rates for each mining method is based on the operational activities cycle time, and mine workers' productivity. The nominal projected performance for LH, shrinkage and R&P stopes are respectively 75, 17.5 and 15 metric tonnes per man-shift. Table 16.23 presents the performance parameter and the production rate for each mining method used.

Table 16.23 – Performance parameter and production rate

Operating Parameters	Units	Long-hole	Shrinkage	Room-and-pillar
Production rate	tpd	400	70	60
Swell Mucking	tpd	NA	28	45
Main Mucking	tpd	90	90	45

16.14 Underground Mine Equipment Selection and Fleet Requirement

Table 16.24 lists the main equipment required for the operation of the mine. Abcourt has the necessary equipment on the Project site and no major purchase is required. To ensure optimal equipment availability, the Project assumes a reconditioning plan.

Table 16.24 – Equipment list

Equipment	Quantity
Mucking Machine LM56	7
Cavo 320 loader	8
Locomotive 3.5 tonnes	11
Wagon 80 ft^3	26
Longtom drill rig	3
Slusher	10
Jacklegs & accessories	46
Stopers	41
Long-hole drilling pneumatic buggy	2

16.15 Mine Personnel

The average mine workforce during production will be 111 employees. The mine will operate on a two-shift schedule of 10 hours a day, 7 days a week. Most employees will work shifts of 10 hours. Others (including management) will work shifts of 10 hours during the day only, on a 4-3 schedule.

Table 16.25 presents the detailed manpower by department including the total workforce on each schedule.

Table 16.25 – Manpower by department

Department	Year - 2	Year - 1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
Underground Services								
Superintendent	0	1	1	1	1	1	1	1
Captain	0	0	2	2	2	2	2	2
Supervisors	0	5	9	10	10	10	10	7

Department	**Year - 2**	**Year - 1**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**
Hoist	0	4	4	4	4	4	4	4
Cage tender	0	4	4	4	4	4	4	4
Platform operators	0	2	2	2	2	2	2	2
Dryer	0	4	4	4	4	4	4	4
Construction services	0	3	3	3	3	3	3	3
Others	0	1	1	1	1	1	1	1
Sub total	**0**	**24**	**30**	**31**	**31**	**31**	**31**	**28**
Maintenance								
Mechanical Services								
Superintendent	0	1	1	1	1	1	1	1
Supervisors	0	2	2	2	2	2	2	2
Underground mechanic	0	6	6	6	6	6	6	6
Surface mechanic	0	2	2	2	2	2	2	2
Machinist	0	1	1	1	1	1	1	1
Welder	0	2	2	2	2	2	2	2
Sub total	**0**	**14**	**14**	**14**	**14**	**14**	**14**	**14**
Electrical Services								
Electricians	0	3	4	4	4	4	4	4
Instrumentation technician	0	1	2	2	2	2	2	2
Sub total	**0**	**4**	**6**	**6**	**6**	**6**	**6**	**6**
Process Plant								
Superintendent	0	1	1	1	1	1	1	1
Supervisors	0	2	2	2	2	2	2	2
Solution operators	0	4	4	4	4	4	4	4
Milling operators	0	2	2	2	2	2	2	2
Crushing operators	0	2	2	2	2	2	2	2
Mechanics	0	4	4	4	4	4	4	4
Metallurgist	0	1	1	1	1	1	1	1
Mineral technician	0	1	1	1	1	1	1	1
Laboratory	0	6	6	6	6	6	6	6
Sub total	**0**	**23**	**23**	**23**	**23**	**23**	**23**	**23**
Technical Services								
Geology								
Chief geologist	0	1	1	1	1	1	1	1
Senior geologist	0	1	2	2	2	2	2	2
Production geologist	0	1	2	2	2	2	2	2

Department	Year - 2	Year - 1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
Production technician	0	1	2	2	2	2	2	2
Geologist (Drill Tracking)	0	2	2	2	2	2	2	2
Drill technician monitoring	0	2	2	2	2	2	2	2
Integration data technician	0	1	2	2	2	2	2	2
Sawyers	0	2	2	2	2	2	2	2
Sub total	**0**	**11**	**15**	**15**	**15**	**15**	**15**	**15**
Engineering Services								
Chief engineer	0	1	1	1	1	1	1	1
Engineer	0	1	2	2	2	2	2	2
Surveyor	0	2	4	4	4	4	4	3
General work technician	0	1	2	2	2	2	2	2
Sub total	**0**	**5**	**9**	**9**	**9**	**9**	**9**	**8**
Environment								
Environmental Coordinator	0	1	1	1	1	1	1	1
Sub total	**0**	**1**	**1**	**1**	**1**	**1**	**1**	**1**
General and Administration								
General manager	0	1	1	1	1	1	1	1
HR & HS superintendent	0	1	1	1	1	1	1	1
HS & training coordinator	0	1	1	1	1	1	1	1
Housekeeping attendant	0	2	2	2	2	2	2	2
Nurse	0	2	2	2	2	2	2	2
Guardians	0	4	4	4	4	4	4	4
Senior accountant	0	1	1	1	1	1	1	1
Sub total	**0**	**12**	**12**	**12**	**12**	**12**	**12**	**12**
Total 4/3	**0**	**8**	**8**	**8**	**8**	**8**	**8**	**8**
Total 7/7	**0**	**86**	**102**	**103**	**103**	**103**	**103**	**99**
TOTAL	**0**	**94**	**110**	**111**	**111**	**111**	**111**	**107**

17. RECOVERY METHODS

The Sleeping Giant mine began operation in 1988 and was last operational in 2014. Since August 2016, the processing plant has been in operation, fed by ore from nearby mines.

Ore is currently processed using two stages of crushing, rod mill grinding, ball mill grinding and a Carbon-In-Leach (CIL) gold plant. A Carbon in Column (CIC) circuit is in place to recover gold in the pre-leach thickener overflow, but it is not in operation as no cyanide is added or is planned to be added in the grinding circuit.

The concentrator flowsheet is shown in Figure 17.1.

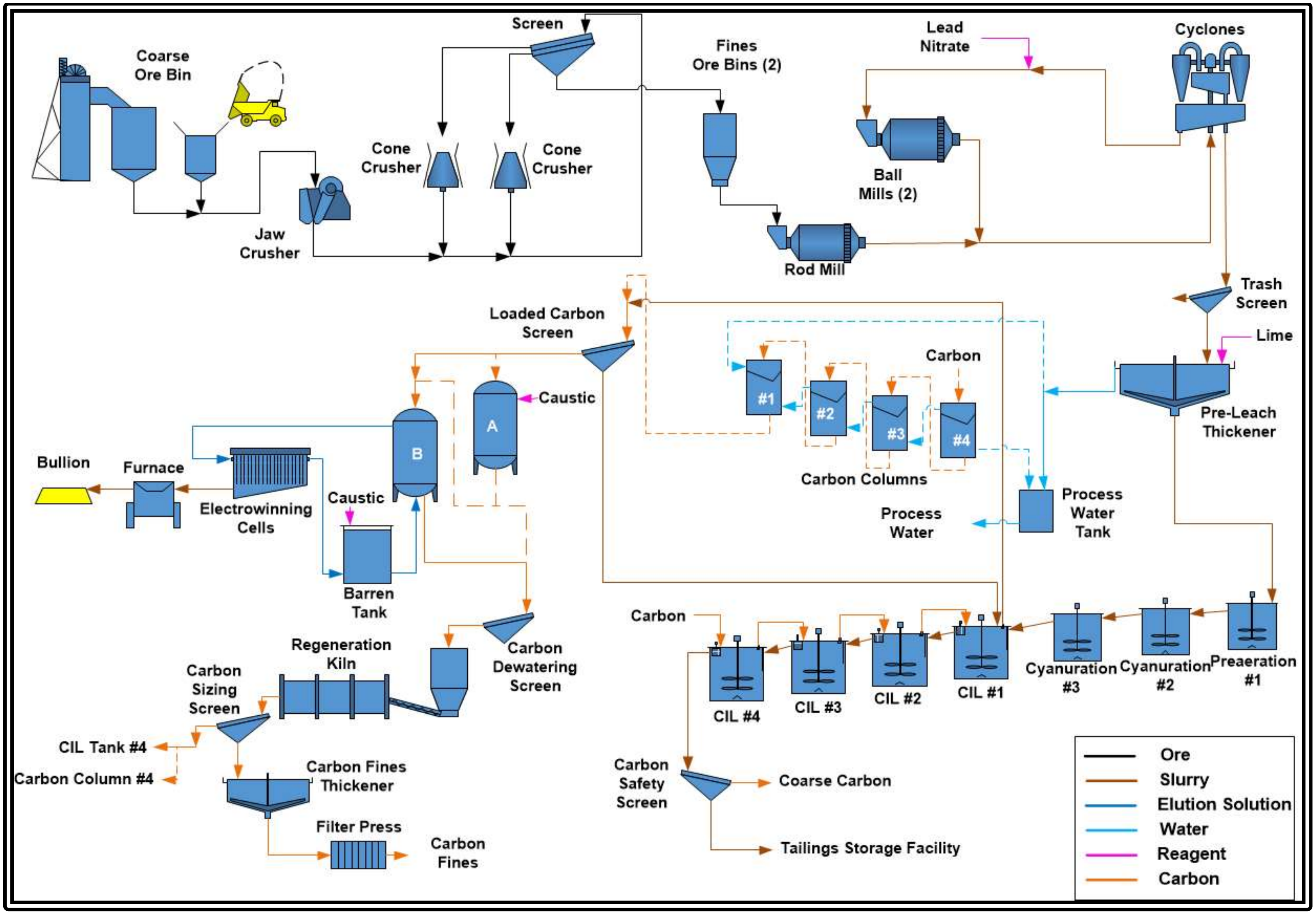


Figure 17.1 – Processing plant flowsheet

17.1 Design Criteria

Table 17.1 summarize the general parameters for the processing plant used for this study.

Table 17.1 – Processing plant major design parameters

Parameters	Value	Unit
Daily Throughput	350	t/d
Head Grade – Au	8.1	g/t
Recovery – Au	96.7	%
Ore Moisture	5.0	%
Pre-Leach Thickener Underflow % solids	54	%
Leach and CIL retention time	46	h

17.2 Process Description

17.2.1 Crushing

The crushing circuit has two (2) stages of crushing in closed circuit with a 6’x20’ double deck vibrating screen.

The coarse ore from the 680-ton bin and the fresh feed from the run-of-mine (ROM) pad are fed to a 24”x36” jaw crusher to reduce the particles size from 45 cm to 10 cm. Product from the jaw crusher and two (2) cone crushers are fed to the double-deck vibrating screen. The fines (≤1 cm) go to two (2) 600-ton fine-ore bins. Material retained by the first deck feeds a 5/2 Standard (STD) cone crusher and the material retained by the second deck feeds a 5/2 Short Head (SH) cone crusher.

17.2.2 Grinding

The grinding circuit has a primary rod mill in open circuit that receive the crusher circuit discharge, and two secondary ball mills in closed circuit with cyclones. The objective of the grinding circuit is to reduce the particle size of the crushed ore.

Crusher circuit product is fed to an 8’x12’ rod mill for a primary grinding before being sent to the cyclones (Linatex HSE-1820 R) for classification. The cyclones’ underflow is sent to two (2) ball mills in parallel for further grinding. The ball mill discharges are also recirculated to the cyclones. The cyclone overflow (D_{80} of 50 µm) is sent to the pre-leach thickener. Lead Nitrate is added in the grinding circuit to counter the negative effects of sulfides and to accelerate the gold dissolution.

17.2.3 Pre-leach thickener and carbon in column

The objective of this circuit is to increase the percent solids of the grinding circuit product before being sent to the carbon-in-leach (CIL) circuit. A carbon-in-column (CIC) circuit is installed to recover gold in the pre-leach thickener overflow, but it is not in operation as no cyanide was added during the latest operation and is not planned to be added in the grinding circuit.

The grinding circuit cyclone overflows feed a 45’ diameter pre-leach thickener. Lime is added to increase slurry pH. The slurry is thickened from 10% to 54% solids. The thickener underflow is sent to Preparation Tank #1. The thickener overflow is sent to the process-water tank. If required, thickener overflow can be sent to the CIC circuit, composed of four (4) carbon columns, to recover gold in solution in the thickener overflow. CIC discharge is sent to the process-water tank. Carbon from CIC tank #1 is sent to the loaded carbon screen.

17.2.4 Leach, CIL and gold recovery

Thickener underflow is sent through one (1) pre-aeration tank, two (2) cyanidation tanks and four (4) CIL tanks. Total leaching time of the leach and the CIL circuits is 46 hours (at 35 t/h, 54% solids). Lime is added to increase slurry pH. Cyanide and air are added to leach the gold. Carbon is added to recover the gold in solution.

The CIL tails from CIL tank #4 pass through a carbon safety screen before being pumped to the tailing storage facility.

Once loaded with gold, the carbon is screened and eluted. The pregnant solution is sent to the gold room for electrowinning, drying, and finally, smelting into gold bullion.

Eluted carbon is regenerated in a kiln and reused in the CIL circuit. Carbon fines generated from the circuit are recovered in bags for further gold recovery.

17.2.5 Reagents and consumables

The major reagents utilized within the process plant are:

- Lime;
- Sodium cyanide (NaCN);
- Caustic soda (NaOH);
- Lead Nitrate;
- Flocculant;
- Descaling agent.

The expected annual consumption for the reagents is presented in Table 17.1.

Table 17.2 – Expected annual reagent consumption

Reagent	Yearly consumption (t/y)
Lime	137
Sodium Cyanide	93
Caustic Soda	47
Lead Nitrate	50
Flocculant	2
Descaling Agent	1

17.2.5.1 Activated carbon

Activated carbon will be fed through the carbon sizing screen from the carbon quench tank. The carbon sizing screen oversize feeds the CIL circuit whereas the undersize fraction feeds the fine carbon circuit.

17.3 Energy Consumption

The annual power consumption for the processing plant is estimated to be 7.4 GWh. The electrical energy requirements for the processing plant were estimated from historical power consumption when processing similar annual tonnage of Sleeping Giant ore.

17.4 Process Water

Process water is used mainly at the grinding area. Water added in the grinding area is partially recovered at the pre-leach thickener and sent to the process water tank. Water from Pond #2 in the tailing storage facilities (TSF) allows for recirculation of water that was sent to the TSF with the CIL tailings. This water is continuously recirculated from the TSF to the processing plant. In case of lack of water in Pond #2, water from the mine water basin could be used to meet the process water needs of the processing plant.

The expected process water requirement at the grinding circuit is estimated between 280 to 330 m^3/d.

17.5 Expected Performances

17.5.1 Recovery

The gold recovery is based on a model that was developed from mill performances. The model predicts the gold recovery based on head grade as described in Item 13.

There is no modification in the plant throughput, equipment size, number of equipment or ore-feed that is expected to change this recovery model.

17.5.2 Throughput

As shown in the historical production data in Item 13 and based on historical daily report when processing Sleeping Giant ore, the processing plant was able to process on a constant basis between 34.0 to 37.9 t/h of ore. The capacity of the processing plant is higher than the current expected processing tonnage of 350 t/d on a 12-hour daily shift. The capacity of the plant is between 700 and 750 t/d when operating 24 hours per day.

18. PROJECT INFRASTRUCTURE

18.1 General Site layout

The content of this section is drawn mainly from Bonneville (2019).

All the infrastructure necessary for the resumption of operations at the Sleeping Giant mine are in place. The treatment plant and tailings pond are still in operation. Since the temporary closure in November 2014, no permanent installations were removed, and the mine was kept de-watered. A weekly inspection is made on the following equipment as if the mine was still operating: the hoist and its components; the production shaft; the dewatering pumps; and, the electrical equipment. There was no addition of infrastructure, and no building were dismantled. The only change concerns the enlargement of the skip in which the trucks dump ore transported from the nearby Elder mine.

A large two-storey building attached to the plant contains the administrative, geology, and engineering offices, the dry house, the offices of the mining department, the workshop, the warehouse, the infirmary, and the mine rescue room.

Shaft #2 and its three-compartment steel headframe, as well as an ore silo and a waste silo with an approximate capacity of 500 and 200 tons each, are located south of the plant. A building for the hoist and the compressor room, the mine water basin, and the waste rock pile are nearby. Shaft #1 is to the east of the plant.

Southwest of the plant, there is a core shack installed in a trailer near the storage yard. Mined ore is also deposited southwest of the plant near the ore pile. There is an electric/hydraulic jackhammer and a bar grate under which a conveyor feeds the plant's jaw-crusher.

The TSF, wastewater treatment unit and the BP-1 polishing pond are northwest of the plant. South of these facilities are the BP-1 and BP-2A basins, as well as the emergency spillway.

The general site layout is shown in Figure 18.1.

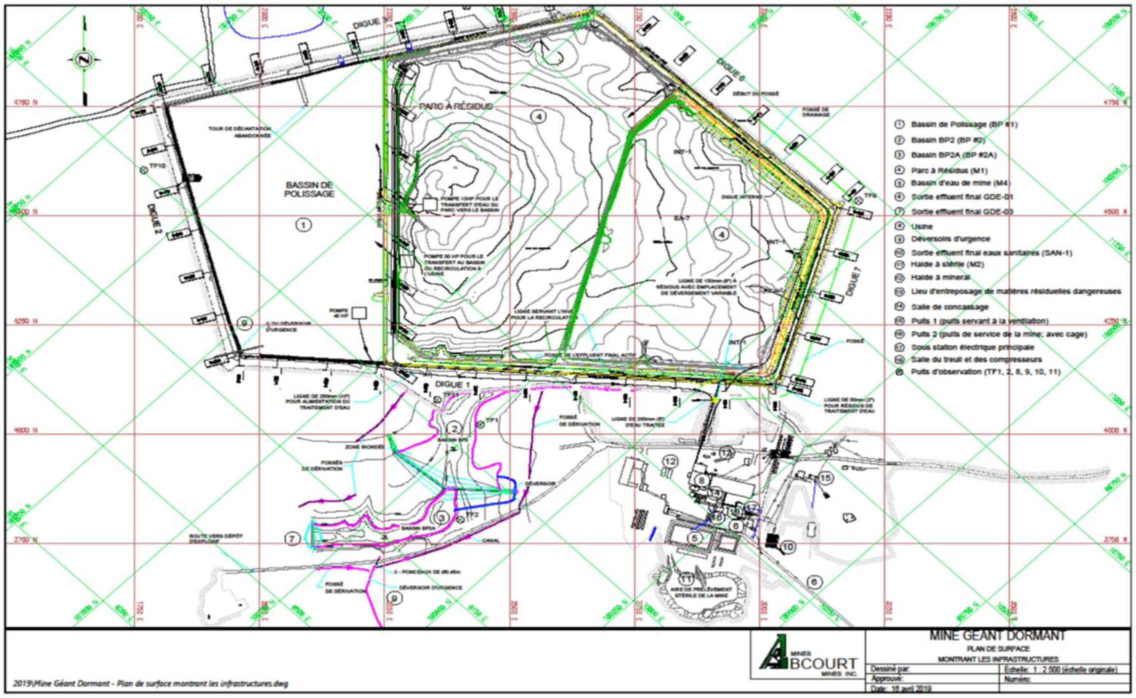


Figure 18.1 – General site layout

18.2 Site Preparation

With the exception of the existing guardhouse, which must undergo a complete renovation, the surface buildings and headframe will not require any major modifications prior to resumption of mining operations at the Project.

The existing pads, located to the east of the plant, are composed of granular material and will be used as foundation for the camp complex. This surface will be prepared by grading and compaction to ensure the stability of the infrastructure, prior to setting up the camp complex. Existing roads and parking areas will also be graded to ensure proper site drainage.

The waste silo and the arc gate located in the silos building, which is annexed to the headframe, will have to be replaced. In addition, the waste chute will have to be rehabilitated.

Some underground infrastructure such as the floors, doors and brattice of the mine service shaft at loading stations 1, 2 and 5, must undergo a complete renovation.

18.3 Site Access Road

Access to the Property is facilitated by paved provincial highway Route 109 that connects Amos to Matagami and passes less than 1 km east of the mine site. The junction of the mine site access road is located near kilometre 118 of Route 109.

18.4 Light Vehicle Roads

Where possible, the reuse of existing roads prepared for forestry operations is prioritized. The network of existing forestry roads allows access to most parts of the Property.

18.5 Mine Site Entrance / Guardhouse

The main site entrance has a guardhouse near the end of the mine-site access road. Guardhouse staff track personnel entering and leaving the mine-site and material deliveries. The guardhouse will need a complete renovation to meet the Project needs. The guardhouse will include a desk with screens for the security cameras and process monitoring, a main fire alarm panel and a restroom. A parking lot is already in place and will be used for visitors and staff, considering most employees will be staying in the camp complex.

The guardhouse layout used as a reference is shown in Figure 18.2.

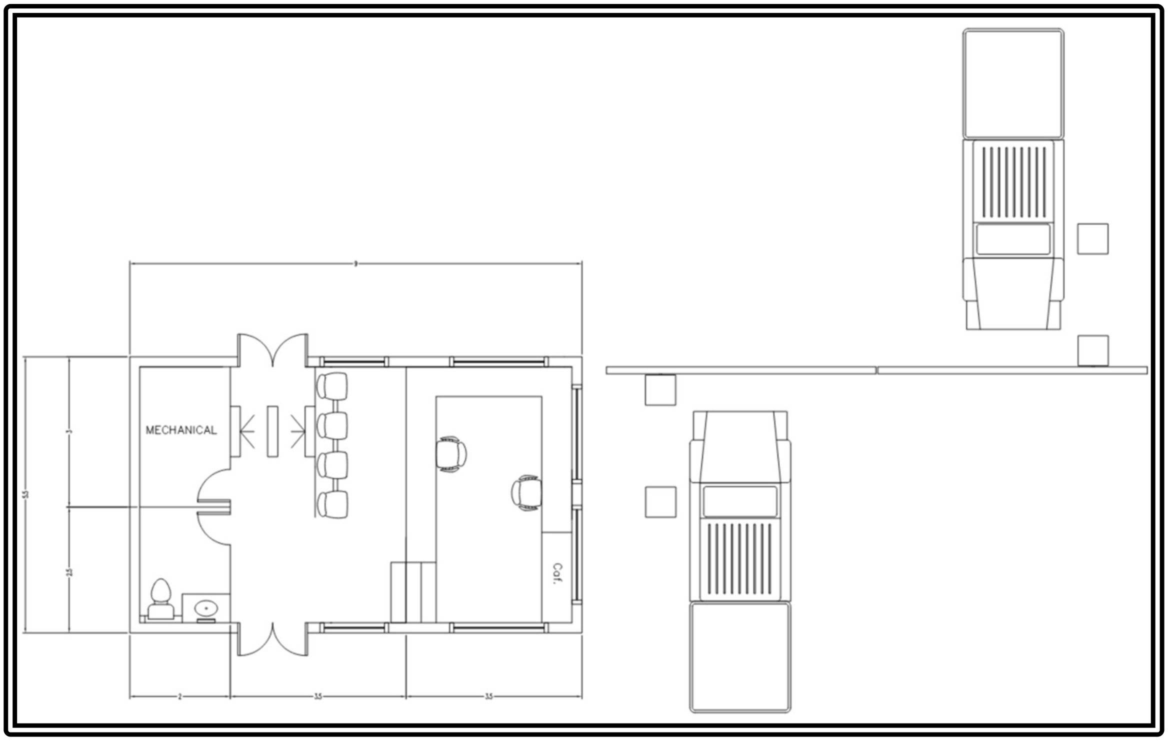


Figure 18.2 – Guardhouse layout

18.6 Surface Electrical Installation, Distribution and Consumption

The mine site is fed from a 25 kV overhead line that is the property of Hydro-Quebec. The tie point is a 25 kV switch located on the steel structure of the outdoor substation. Beyond that point, three single phase transformers, each of 2.5 MVA, are arranged in star-delta forming a three-phase system with an ungrounded 2.4 kV secondary side. Fuses protect the transformers on the 25 kV primary side and the 2.4 kV feeders on the secondary side. There is no ground fault protection on the 2.4 kV distribution.

2.4 kV is used as the main distribution voltage for the entire site. It covers the following areas: hoist, underground mine, crushing plant, concentrating plant, administration building, and tailings park. The utilized voltage in those areas is mainly 600 Volts and 120/208 Volts.

Electrical distribution on-site has improved, and it is now safer than it was originally. The system evolved from an ungrounded system to a solidly grounded system and to a resistance grounded system. A resistance grounded system has the advantage of limiting the ground fault current, which is the most common type of fault, and thus limiting the ground potential rise (GPR) which in turn minimizes the risk of electrical shock.

The Canadian standard “Use of electricity in mines, M421-16 (reaffirmed 2021)” has a rule stating that “The ground potential rise (GPR), at movable or mobile equipment shall not exceed 100 V” (article 6.8.3: Ground potential rise).

Mobile electrical equipment is designed to be energized while moving. Movable electrical equipment don’t have grounding networks and are designed to be moved only when de-energized.

The use of electric powered mobile equipment is common in mining. There are many 2.4 kV and 600 V components present in the mine and people are exposed to it. For that reason, it is common practice to have ground fault protection in electrical distribution to protect the personnel and to limit damage to equipment.

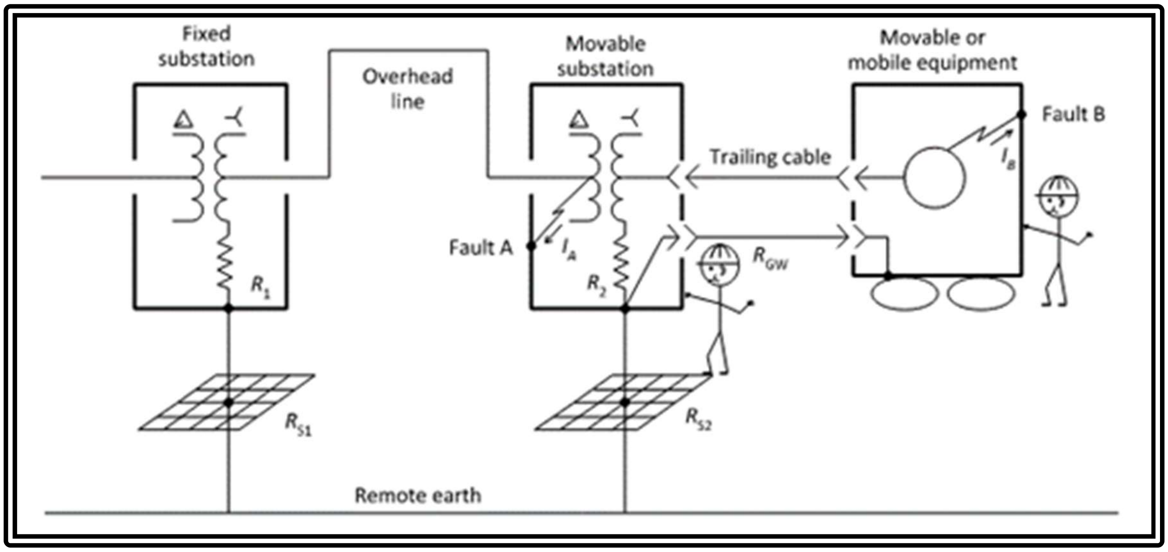


Figure 18.3 – Ground potential rise and ground fault voltage

To provide ground fault protection at 2.4 kV, modifications must be made in the main sub-station to add and replace many electrical components, e.g., breakers, transformers, grounding resistors, electrical room, switchgear etc. The actual surface single line diagram is presented in Section 27.

Three (3) options to complete the modifications were evaluated and are presented as follows:

Option 1 would modify primarily the mine underground feeder and involves adding a 25 kV breaker on the transformer bank primary, a zig zag grounding transformer on the

2.4 kV main bus, an isolating 2.4-2.4 kV Delta-Wye transformer with a resistance grounded neutral and a 2.4 kV switchgear.

Option 2 would modify the outside line-up of 2.4 kV open-air fused switches. This setup was originally custom made at the site and has the following shortcoming: it is not protected from precipitation; it cannot be remotely operated in case of ground fault; and, it poses a safety hazard to operate and to maintain because of the proximity of the electrical live parts. This legacy setup would be replaced by a 2.4 kV MCC. Also included in this option, is a 25 kV breaker on the transformer bank primary, a zig zag grounding transformer on the 2.4 kV main bus and zig zag grounding transformer for the 2.4 kV emergency generator.

Option 3 would replace the Wye-Delta 7.5 MVA transformer bank (ungrounded on the secondary side) with two (2) 4 MVA Delta-Wye transformers, each having a resistance grounded neutral. Each transformer feeds a 2.4 kV MCC to replace the outside line-up of 2.4 kV open air fused switches. Also included in this option, is a 25 kV breaker on each transformer primary and a zig zag grounding transformer for the 2.4 kV emergency generator.

Option 2 is preferred but detailed engineering is required to show practicality.

18.6.1 Underground mine electrical substations

As stated in Item 18.6, the mine’s 2.4 kV electrical distribution is ungrounded. It represents an electrical shock hazard on the casing of each transformer installed in the underground electrical room. Those transformers, are used to lower the voltage to utilisation level for different purposes e.g., mine ventilation, mine dewatering, power take-off stations, lighting and services transformers, mobile mining drills and bolters, etc.

Each underground 2.4-0.6 kV transformer should be equipped on its 600 V side with a grounding resistor and breakers, each having ground fault protection (one for each circuit).

During the site visit, such ground fault equipment was only observed at level 1060. Electrical substation at all other levels would have to be either modified or simply converted using a new skid-mounted transformer and 600 V distribution.

18.6.2 Emergency generator

The onsite 800 kW at 2400 V emergency generator is primarily used as a backup for powering the underground mine dewatering pumps and the tailings parc pumping station. For unknown reasons, the generator stator suffered major failure and needs to be rewound. The status of the rotor is unknown. The generator is of the synchronous type, which makes it a more complex machine that required controls for synchronizing the field excitation and breaker closing. The field excitation controls and the protection relays are outdated and need to be replaced.

18.6.3 Camp

Abcourt wants to add a 125-worker camp at their site to accommodate its labour, but the main electrical substation transformers can’t supply this additional load. To overcome this, a split would be added on the 25 kV overhead line to create a new branch before it reaches the main substation. The newly created branch would feed the new camp site and the other one would feed the existing surface main electrical substation. The Hydro-Quebec metering station would be relocated ahead of the new branch to monitor consumption of the entire mine site.

18.7 Administration Building and Dry Complex

The administrative offices are located in a large two-storey building attached to the plant. These offices are in a favorable state for the resumption of mining activities. In addition to these administrative offices, this building also contains the geology and engineering offices, the dry house, the offices of the mine department, the mechanical and electrical workshop, the warehouse, the infirmary, and the mine rescue room.

All of these premises are in good condition and are operational. They currently do not require any major improvements for the resumption of mining activities at the Project.

18.8 Maintenance Shop and Warehouse

The mechanical and electrical maintenance workshop is also located in the two-storey building attached to the plant. This maintenance workshop is used for the maintenance of all underground equipment and has an overhead crane to facilitate operations. In the same building is a warehouse for the storage of various spare parts for the proper functioning of the equipment. These facilities are in good condition, functional and ready for the resumption of operations.

An outdoor storage area is located southwest of the crushing facilities and near the ore piles. It has a shelter that can provide protection for the equipment from bad weather and is currently in acceptable condition.

18.9 Camp complex

The camp complex will be made up of used prefabricated modular-type units or “blocks” installed on steel tripods and will include: dormitories, cafeteria and dining room. Each block will be supplied with the required potable water, sewage, electricity, propane, and fire water (cafeteria and dining block only). Each block will include propane gas-fired heating units and water heaters with storage tanks and recirculation pumps.

The layout of the dormitory aisle number 100 is shown in Figure 27.6. There will be three, one-storey rows of dormitories consisting of six modules each, for a total of 86 individual rooms and 14 double rooms. The connecting corridor of each dormitory will include a laundry room, a washroom, a housekeeping room, a mechanical/electrical room, and a living room.

The cafeteria and dining room will allow for food-storage, -preparation and -service for on-site personnel. This area will also include a hand wash station, a coat room, a washroom, offices, garage doors for the receiving area, a mechanical room, and a refrigerated room. The cafeteria and dining room area will be equipped with sprinklers and a commercial exhaust hood with fire protection for safety.

18.10 Communications and IT

There are two underground communication systems at the Sleeping Giant mine. One is a hard-wired telephone system that provides voice communication between the mine office, hoist room, shaft building, shaft stations, mess halls and other key locations. The second is a radio system that utilizes an underground network of radiant cables that communicate with surface antennas. It is used for voice communication from almost all underground and surface locations. It is not used for transmitting data such as fan status, air quality, pump status and water level in sumps.

The site has high-speed internet access.

18.11 First Aid / Emergency Services

The Sleeping Giant mine has a complete and functional infirmary to provide first aid to its employees when necessary. The infirmary and the mine rescue room are located above ground in the two-storey building attached to the plant. All underground shelters are fully equipped with stretchers and first-aid kits, in case of emergencies. If more advanced medical attention is required, local paramedical services will be called upon. Under these conditions, the hospital and the Community Health Centre (Centre de santé et de services sociaux) in Amos are the closest medical centres and are located seventy-seven kilometres from the property.

18.12 Explosives Plant and Storage

The Sleeping Giant mine has warehouses on the surface for the storage of explosives, but they are currently out of service due to the absence of on-site operations. Explosives and detonators are stored underground until mining operations resume. Each level underground has a detonator chamber and an explosive chamber. All these warehouses have a storage permit obtained from the Sureté du Québec. This entity periodically checks the facilities to ensure compliance with effective standards.

18.13 Fuel Storage and Delivery

There is an on-site diesel storage tank with a capacity of 10,000 litres that is surrounded by concrete barriers for protection. This tank is located above ground and is inspected annually to ensure optimal and safe operation, as well as to maintain the user license.

18.14 Site utilities

18.14.1 Fire water and system

The mine has a Viking fire protection system installed to ensure the safety of surface facilities. This fire protection system is present at the crusher, conveyors and in the headframe building. This system is powered by an automatic 1500 gallon per minute diesel pump, which is designed to start itself in order to balance the pressure in the system when required. The fire water system is functional and periodically verified to ensure that it is in proper working order.

18.14.2 Potable water

Drinking water is distributed in various buildings in the Sleeping Giant mine complex through a piping system.

Based on the information available, the drinking water is currently provided by two 20 cm diameter 20 m deep wells located approximately 800 m southeast from the mine site. The water is pumped by two submersible pumps (one in each well) and directed to an aboveground storage reservoir. The current reservoir is located south of the headframe. A water treatment system is not present, as treatment is not required.

This water is also used to supply water for washrooms, the different buildings (guardhouse, processing plant, dry house, etc.). Additional testing will be required to confirm that the current well installation could supply the future camp complex planned for 125 workers.

18.14.3 Sewage treatment

The domestic wastewater is currently directed through a sewage piping system to a septic tank and an intermittent sand filter. The effluent is directed to a draining ditch/small creek located southeast of the filter.

Considering that the number of workers staying on-site will increase (125 workers on a 7-day rotation schedule, 355 days per year), this will generate an increase in wastewater volume, which the current system is not designed to receive. The design of a new wastewater treatment system is required to accommodate the increased volume of wastewater to treat. A ministerial authorization from the Ministère de l’Environnement, de la Lutte contre les changements climatiques, de la Faune et des Parcs (MELCCFP) will be required for the modification of the wastewater treatment system.

18.15 Waste Stockpile

Waste rock from the underground mine is stored in the waste rock pile located south of the headframe. In the recent years, waste rock was used for the internal dike lifts related to cell 1 of the TSF.

A new cell (2A) is planned to be constructed at the TSF. A recent survey shows that there is enough waste rock present at the waste stockpile for the construction of cell 2A. Additionally, new waste rock will be generated during the future mining operation. According to the design of the cell 2A, this volume of waste rock will be sufficient for the construction of the cell.

18.16 Mineralized Material Stockpile

An ore pad was constructed in 1992-1993. The ore pad is located on the southwest side of the processing plant. There is an electric/hydraulic jackhammer beside and a bar grate under which a conveyor feeds the plant's jaw crusher. The size of the ore pad is approximately 30 m x 60 m for a capacity of 12,000 tons.

18.17 Process plant complex

The gold extraction and concentration process involves a series of steps including crushing, grinding, gravity separation, cyanidation, and refining.

The processing plant does not require additional mechanical equipment to resume operations at the expected feed tonnage to recover gold at the expected recovery as confirmed by Guy Comeau (QP for Item 17). Significant maintenance is required for the crushers, tanks and silos. Conveyors, pumps, crushers and cranes will require normal maintenance to ensure their longevity. This maintenance work must be performed before the plant resumes its operations. This will minimize the risk of unplanned breakdowns that could result in production losses.

18.18 Mining Wastewater Management

Several hydraulic and hydrological studies have been carried out in the past by various engineering firms for the Sleeping Giant TSF. The studies and analyses have shown certain issues related to the tailings and mining wastewater management on the site within a medium to long term production. The water balance for future years of operation shows that the operating mode of the TSF is at risk, because the facility cannot contain enough water to ensure operations while maintaining a sufficient volume available to manage the design flood according to Directive 019.

The current mitigation methods implemented by Abcourt are effective in mitigating operational problems. It was possible to meet the water needs of the processing plant each year by using water from the mine water basin during the last winter months of the past winters. On the other hand, in the event that the TSF had to manage the project flood as defined in Directive 019 of the MELCCFP, the reaction time to set up a pumping system to avoid an overflow could be too long and put at risk the TSF operation. In order to reduce operational and environmental risks, the available volume for the mining wastewater should be increased.

In order to increase the retention volume of the TSF, dredging works will be carried out in Cell 2. Operational expenditure (OPEX) costs for dredging works are planned for the first four (4) years of operation.

Regarding surface waters, draining works have been done on certain parts of the Property and the TSF. The goal was to divert clean runoff water to outfalls downstream of the Property.

18.19 Process Mining Wastewater

Mining wastewater comes primarily from the mining tailings. The mining wastewater treatment train is an annual batch process. The batch begins at snowmelt in the spring and ends late in the fall when Cell 3 is lowered in preparation to increase the volume to manage the spring snowmelt and the design flood according to Directive 019. The mining wastewater is treated to decrease the concentration of metals (mainly copper) and cyanides. The processing unit is located near Cell 3, which is also used as a treatment and polishing basin. The treatment chain is as follows:

- Cell 3 is used as a treatment and polishing basin;
- A piping system allowing continuous injection and dosing of ferric sulphate during recirculation;
- 2 pumps are used to recirculate water from Cell 3 through nozzles (fountains) installed over Cell 3 prior to being discharged. Nozzles at the discharge point in Cell 3 are used to help reduce the concentration of cyanides by natural attenuation;
- BP-2 polishing pond: when the quality criteria are met, water exits from Cell 3 to BP-2 which is used for polishing treatment when needed;
- BP-2A polishing basin: used as a sampling point for the analysis of treated water before being discharged into the environment.

On an annual basis, the treatment in the TSF begins in May-June and ends in October. The current wastewater treatment train will be maintained. For the resumption of operations, maintenance work or replacement is planned for the pumps and nozzles currently used, to allow for a better treatment efficiency.

Water from the underground mine workings is pumped to the surface to keep the mine dry. According to information available, excess water from the mine water basin that is not reused in the gold process, is discharged to the environment. No treatment is required according to the monitoring program.

18.20 Tailing Storage Facilities

The current TSF is located north of the processing plant and is made up of three cells: Cell 1 (deposition pond), Cell 2 (recirculation pond) and Cell 3 (polishing pond). Dikes 1, 2, 3, 6 and 7 surround the TSF, whereas inner dike 1 and the median dike separates the three cells. An access road dividing Cell 2 was built in 2019. An emergency spillway is located at the western end of the TSF, in cell 3. It should be noted that for previous years, Cell 1 has been raised in 4 successive lifts to reach the elevation of lift 5 to manage tailings deposition.

Figure 18.4 presents an overview of the tailing storage facilities.

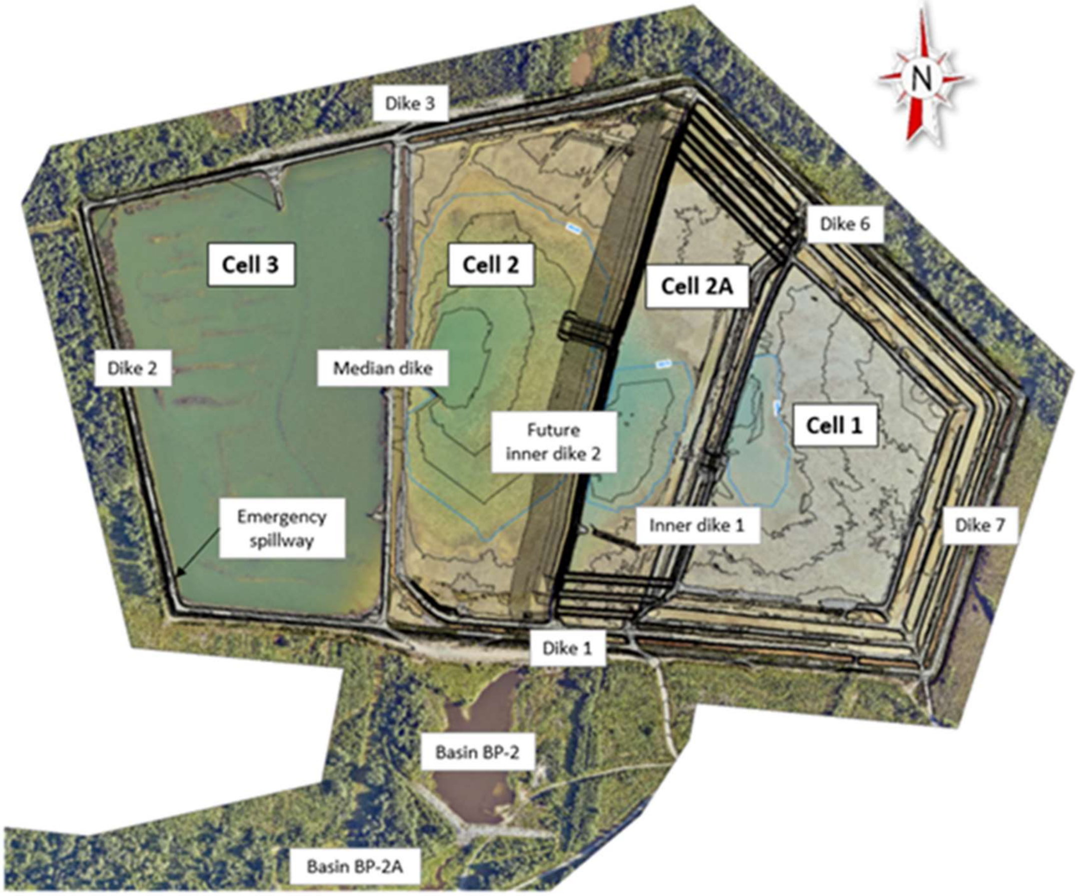


Figure 18.4 – Tailing storage facilities overview

Based on current data, the space available for tailings deposition in Cell 1 is approximately 24,000 m^3. To allow the storage of all the tailings planned according to the mine production plan of the Project, modifications must be made to the TSF. It is proposed to split cell 2 into two parts. This modification would allow the storage of the mine tailings in the future cell 2A and the remaining Cell 2 would continue to be used for recirculation of mining process water to the processing plant. Cell 2A will be built by raising dikes 1 and 6 by lifts similar to those in Cell 1, and by building an inner dike 2 at the location of the existing access road in Cell 2, using methods to optimize storage space of Cell 2A. The anticipated tailings storage capacity of the future cell 2A is estimated at 500,000 m^3.

The raising of dikes 1, 6 and inner dike 2 in successive lifts will be carried out according to the requirement of the mine production plan. Three successive lifts for cell 2A are planned to contain the tailings of the mining plan (720,000 tons). The lifts of the various dikes will be built using tailings and waste rock.

Table 18.1 presents the planned annual construction work related to Cell 2A. According to this scenario, cell 2A will need to be raised each summer for the first, second, third and fourth year to store the mine tailings anticipated, as well as the mine tailings dredged from cell 2 during the 3rd year of production.

Table 18.1 – TSF: Construction Staging - Year 0 to 7

Year	Cumulative total mine tailings produced (m3)	Dredging of cell 2 (m3)	Cumulative total space available for tailing storage (cells 1 and 2A), m3 *	Lifts to be constructed Inner dike 2
0	-	-	24,000	-
Yr 1	12,843	-	182,006	Lift 2
Yr 2	82,051	-	302,788	Lift 3
Yr 3	159,288	110,000	397,592	Lift 4
Yr 4	238,962	-	524,000	Lift 5
Yr 5	317,850	-	524,000	-
Yr 6	397,524	-	524,000	-
Yr 7	450,148	-	524,000	-

*Does not take into account the volumes of dry tailings excavated from cells 1, 2A and 2 in order to build the different lifts of inner dike 2 and dikes 1 and 6.

19. MARKET STUDIES AND CONTRACTS

In January 2015, Abcourt, Auramet International LLC and the Royal Mint signed an agreement under which all of Abcourt's precious metals refined by the Royal Canadian Mint were transferred to Auramet International LLC as part of the sale process. As of the date of this report, that agreement is still in force.

The 2023 PEA assumes that the Project will produce gold in the form of gold doré bars. The market for gold doré is well established and accessible to new producers.

19.1 Market Studies

No market studies have been conducted by Abcourt or its consultants in relation to the gold doré that would be produced by the Project. Terms and conditions included as part of the sales contracts are expected to be typical for this commodity. Gold is bought and sold on many markets in the world, and it is not difficult to obtain a market price at any time. The gold markets are very liquid with many buyers and sellers active at any given time.

19.2 Assumptions

The long-term price of gold and exchange rates were estimated based on discussions with experts, trailing averages, consensus analyst estimates, and recently published economic studies that were deemed to be credible. For the Report, a gold price of $1,650 USD/oz and an exchange rate of 0.77 USD:CAD were used. Table 19.1 outlines the refining and pricing assumptions used in the economic analysis as described in Chapter 22. Figure 19.1 show the historical monthly average price for gold since May 2020.

Table 19.1 – Refining and pricing assumptions

Assumption	Unit	Value
Gold payable	%	100
Doré refining cost (including deductions, transportation and insurance costs)	USD/oz	5.00
Royalty payment	%	2
Gold price	USD/oz	1650

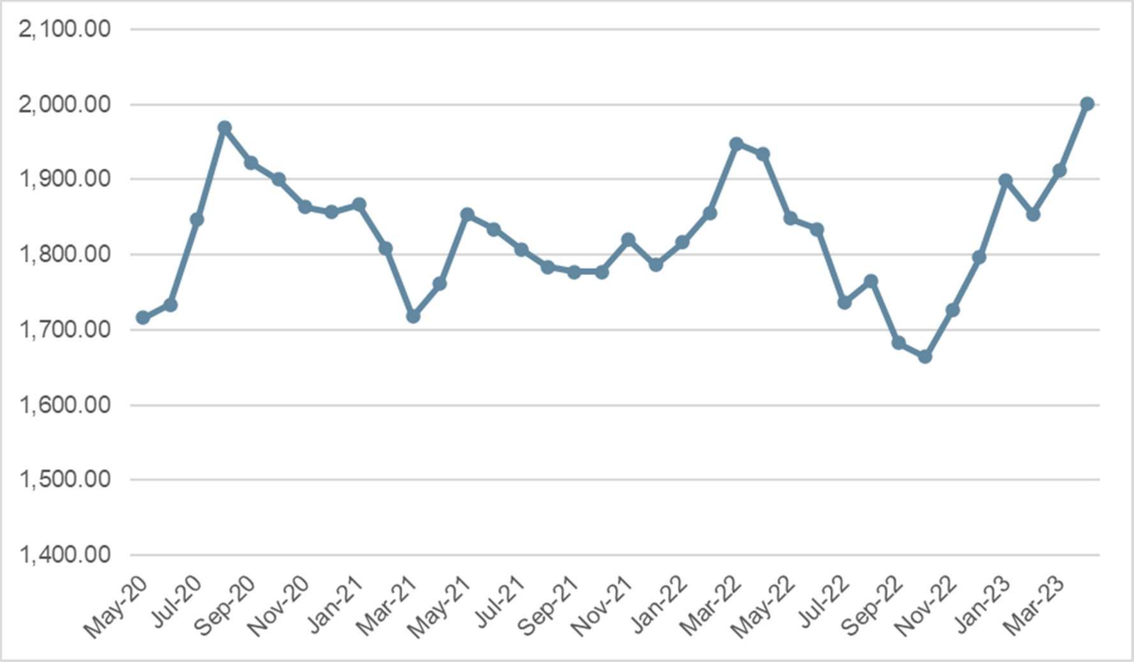


Figure 19.1 – Historical average monthly gold price (USD/oz)

The forecasted exchange rate and precious metal prices are kept constant and are meant to reflect long-term expectations over the life of the Project. It should be noted that the exchange rate and precious metal prices can be volatile and that there is the potential for deviation from the life-of-mine forecasts.

19.3 Royalties

Based on local mining convention, a royalty of 2 % has been assumed for the Project.

19.4 Contracts

19.4.1 Gold Doré

There are no refining agreements, sales contracts, or other contracts currently in place that are relevant to the Report. The doré produced by the Project will be shipped to a precious metals refinery for recovery of the gold into high purity bars meeting the minimum London Bullion Market Association (LBMA) delivery standards.

19.4.2 Other Contracts

As of the effective date of this Report, Abcourt has not signed any agreements or services contracts as part of the Project.

20. ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental baseline studies

20.1.1 Site history

In the late 1950s, many exploration surveys were conducted in the Sleeping Giant area. After gold mineralization was discovered in the late 1970s and early 1980s, there followed four phases of commercial production at the Sleeping Giant mine (Services Miniers PRB, 2019).

There were several changes in ownership prior to Abcourt's acquisition of the project from Aurbec in 2016.

The Sleeping Giant mine was in production from 1988 to 2014. Since the shutdown in 2014, dewatering operations have been maintained and all the infrastructures were kept operational. No addition or dismantling of infrastructures has occurred.

Ore from Abcourt's nearby Elder mine was processed at the Sleeping Giant mill from 2016 to July 2022. Ore from the nearby Vezza mine was processed at the mill for a brief period ending in 2014, and once again in 2018.

The current mining plan of the Project consists of extracting 720,000 tons of ore, from existing and new stopes that can be accessed almost exclusively from existing levels.

20.1.2 Ongoing activities and studies

Sleeping Giant mine is subject to Canadian and Québec requirements in term of monitoring studies and environmental controls. The mine site is subject to the federal metal and diamond mining effluent regulations. Monitoring studies are conducted routinely to identify potential effects caused by effluents on the aquatic habitat. In order to comply with the regulations, environmental effects monitoring (EEM) was performed in 2005 (cycle 1), 2008 (cycle 2), 2010 (cycle 3), 2013 (cycle 4), 2016 (cycle 5), and 2019 (cycle 6). Cycle 7 is currently underway (GCM Consultants; AJ Environment, 2021).

As part of the regulations respecting industrial depollution attestations, Abcourt is also required to produce an annual report on its activities. The most recent report covers year 2021 and summarizes monitoring of effluents, groundwaters, emissions of atmospheric contaminants, waste production, and tailings management (GCM Consultants, 2022).

20.1.3 Planned activities

Environmental baseline studies on the Project have not been carried out but may be required. If necessary, these studies would cover the footprint of the new infrastructure. To date, anticipated changes to the Project include the addition of a new electrical entrance cabling network to the site, additional worker camps and domestic wastewater treatment, adapted as required. Lastly, modifications to the configuration of the existing TSF cells will also be required to extend the service life of this infrastructure.

20.2 Regulatory context

In Québec, the procedure for environmental assessment is included in the Environment Quality Act (EQA, Chapter Q-2) and its regulations.

20.2.1 Environmental impact assessment procedure

With its location, the proposed Project is included in the southern procedure regarding the environmental evaluation process. As listed in Part II of Schedule I of the Regulation respecting the environmental impact assessment and review of certain projects (Chapter Q-2, r. 23.1), mining projects are notably subjected to the procedure when:

- 22 (2) the establishment of a mine whose maximum daily capacity for extracting any other metal ore is equal to or greater than 2,000 metric tons;
- 23 (1c) the construction of a treatment plant for any other metal ore whose maximum daily treatment capacity is equal to or greater than 2,000 metric tons;
- 23 (2) any increase of the maximum daily treatment capacity of a plant referred to in subparagraph c (or d) of subparagraph 1 of the second paragraph to reach or exceed, as the case may be, any of the treatment thresholds provided for therein;

As the proposed Project remains below the thresholds mentioned in the EQA, it does not trigger the environmental impact assessment procedure. The Project is also below the thresholds of the federal legislation (*Impact Assessment Act and Physical Activities Regulations*) and as such, does not trigger a federal impact assessment study; however, the proposed Project is subjected to a permitting process and must go through an authorization process (see Item 21).

20.2.2 Permitting process

20.2.2.1 Québec process

The proposed Project falls under the EQA and the *Regulation respecting the regulatory scheme applying to activities on the basis of their environmental impact* (Q-2, r. 17.1).

The permitting process is triggered by the EQA Section 22, which specifies the following:

Subject to subdivisions 2 and 3 of the Regulation, no one may, without first obtaining an authorization from the Minister, carry out a project involving one or more of the following activities:

- (2) any withdrawal of water, including related work and works, to the extent provided for in Division V;
- (3) the establishment, alteration or extension of any water management or treatment facility referred to in section 32, and the installation and operation of any other

apparatus or equipment designed to treat water, in particular in order to prevent, abate or stop the release of contaminants into the environment or a sewer system;

- (4) any work, structures or other intervention carried out in wetlands and bodies of water referred to in Division V.1
- (6) the installation and operation of an apparatus or equipment designed to prevent, abate or stop the release of contaminants into the atmosphere.

The permitting process is also triggered by the Q-2, r. 17.1 for which triggers include:

The following mining activities require authorization pursuant to subparagraph 10 of the first paragraph of Section 22 of the Act:

- (1) the excavation of ramps, sinking of shafts, and any other excavation work performed to extract ore or explore for mineral substances;
- (2) any activity carried on in connection with ore extraction;
- (3) any activity carried on in connection with ore processing;
- (4) the management of mine tailings, including the establishment and operation of a mine tailings site;
- (5) the management of wastewater from mines, including the establishment and operation of the infrastructures needed for that purpose;
- (6) the storage of ore or concentrates, including the establishment of accumulation areas and the crushing and sieving of ore and concentrates;
- (7) the construction of capping during site rehabilitation and restoration and any work that can alter or modify previous rehabilitation work on an accumulation area for mine tailings.

Complementary mining activities such as atmospheric emissions and works in wetlands are also triggers of the permitting process.

Every applicant must submit its proposed project by following the procedure as stated in Q-2, r.17.1, paragraphs 16, 17, 18 for general content and paragraphs 79 and 80 for mining activities. In summary, the data required includes information on the applicant, description of the project and its activities, the impacts of the project, and all the plans, specifications, and studies needed in support of the application.

20.2.2.2 Current Certificates of Authorization

In June 2016, the transfer of Certificates of Authorization (CofA) from previous owners to Abcourt were completed.

The transfer of CofA to Abcourt includes:

- 7610-10-01-70049-24 (401349775)
    - Exploitation of the mine and operation of a processing plant
- 7610-10-01-70049-26 (401349771)
    - Exploitation of a water treatment plant of the tailings
- 7610-10-01-70049-28 (401349778)
    - Construction of internal dykes in the tailing pond.
- 7610-10-01-70049-29 (401349732)
    - Processing gold ore from Vezza property at the Sleeping Giant mining complex. This was ore processed at the Sleeping Giant plant from a nearby mine site operated by another mining company. This mine was closed in 2018, so no more ore from this site is being processed.
- 7610-10-01-70049-30 (401349776)
    - Processing gold ore from Elder property at the Sleeping Giant mining complex. There has been no activity at the Elder site since July 2022, so no more ore is being processed at the Sleeping Giant mining complex.

Apart from these key CofA, other CofA related to operations at the mine site are also in effect, including those related to septic installations, drinking water supply, storage of hazardous wastes, and exploitation of a sandpit; however, from the standpoint of the proposed Project, the CofA related to drinking water, wastewater treatment and the TSF may need to go through a modification process to reflect the changes that the Project could command.

The mine site is also subject to a depollution attestation under Division III of the EQA. Technical advice received from the MELCCFP in March 2020 related to the depollution attestation mentioned that new monitoring wells to assess the groundwater quality is required, considering that some values are exceeding applicable criteria in monitoring wells that were improperly installed.

20.2.2.3 Permitting requirements

Throughout all stages of the Project (i.e., construction, operations, and closure), activities conducted will be required to comply with provincial and federal acts and regulations. The detailed engineering and operations will consider the conditions, mitigation measures and monitoring requirements associated with the authorization process. It shall also consider all applicable environmental standards included in other relevant provincial acts, regulations, guidelines, and policies. The most relevant for which permits may be required are as follows:.

20.2.2.3.1 Provincial (Québec) jurisdiction

The following are examples of Main Provincial Acts, Regulations and Guidelines applicable for mining activities that may be required by the provincial jurisdiction.

Mining Act (M-13.1)

- Regulation respecting mineral substances other than petroleum, natural gas and brine (M 13.1, r. 2)

Environmental Quality Act (Q-2)

- Regulation respecting activities in wetlands, bodies of water and sensitive areas (Q-2, r.01)
- 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. 7.1)
- Regulation respecting compensation for adverse effects on wetlands and bodies of water (Q-2, r. 9.1)
- 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 the burial of contaminated soils (Q-2, r. 18)
- Regulation respecting the landfilling and incineration of residual materials (Q-2, r. 19);
- Regulation respecting waste water disposal systems for isolated dwellings (Q-2, r. 22)
- 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 quality of the atmosphere (Q-2, r. 38)
- Regulation respecting the quality of drinking water (Q-2, r. 40)
- Regulation respecting the charges payable for the use of water (Q-2, r. 42.1)

Threatened or Vulnerable Species Act (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)

Watercourses Act (R-13)

- Regulation respecting the water property in the domain of the State (R-13, r. 1)

Sustainable Forest Development Act (A-18.1)

- Regulation respecting the sustainable development of forests in the domain of the State (A-18.1, r. 0.01)

Conservation and Development of Wildlife Act (C-61.1)

- Regulation respecting wildlife habitats (C-61.1, r. 18)

Lands in the Domain of the State Act (c. T-8.1)

Building Act (c. B-1.1)

- Construction Code (B-1.1, r. 2)
- Safety Code (B-1.1, r. 3)

Explosives Act (E-22)

- Regulation under the Act respecting explosives (E-22, r. 1)

Cultural Heritage Act (P-9.002)

Highway Safety Code (C-24.2)

- Transportation of Dangerous Substances Regulation (C-24.2, r. 43)

Occupational Health and Safety Act (S-2.1)

- Regulation respecting occupational health and safety in mines (S-2.1, r. 14)

Dam Safety Act (S-3.1.01)

- Dam Safety Regulation (S-3.1.01, r. 1)

Directives and Guidelines

- Directive 019 sur l'industrie minière (translated into Directive 019 on the mining industry; MDDEP, 2012)
- Lignes directrices relatives à la valorisation des résidus miniers (translated to Beneficial reuse of mine tailings guidelines; MDDELCC, 2015)
- Guide de préparation du plan de réaménagement et de restauration des sites miniers au Québec (translated to Guidelines for preparing mine closure plans in Québec; MERN, 2022) related to the preparation of mine closure plans in Quebec
- Guide d'intervention – Protection des sols et réhabilitation des terrains contaminés (translated to Intervention guide - Soil protection and rehabilitation of contaminated sites; MELCC, 2021)
- Guide de caractérisation des résidus miniers et du minerai (translated to Mine tailings and ore characterization guide; MELCC, 2020)
- Analyse de risques et de vulnérabilités liés aux changements climatiques pour le secteur minier québécois (translated to Climate change risk and vulnerability analysis for the Quebec mining sector; URSTM, 2017).

20.2.2.3.2 Federal (Canada) jurisdiction

The following are examples of Main Federal Acts, Regulations and Guidelines applicable for mining activities that may be required by the federal jurisdiction.

Fisheries Act (R.S.C., 1985, c. F-14)

- Metal and Diamond 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)

Directives and Guidelines

- Environment Canada Environmental Code of Practice for Metal Mines (EC, 2009)
- Guidelines for the Assessment of Alternatives for Mine Waste Disposal (EC, 2016)
- Strategic climate change assessment (ECCC, 2020).

20.3 Social and community considerations

20.3.1 Communities and main characteristics

The study area is located in the southern part of the territory of the Eeyou Istchee James Bay Regional Government on Category III Lands.

The Regional Government is governed under the laws of Québec and exercises the jurisdictions, functions, and powers over Category III Lands in the Eeyou Istchee James Bay Territory. The Regional Government has the authority to affirm its jurisdiction as a regional county municipality (“MRC”). It also acts as a regional conference of elected officers. The Regional Government is directed by a Council composed of 11 Cree representatives, 11 Jamésien representatives and one non-voting representative of the Gouvernement du Québec.

The Regional Government is responsible for managing Category III Lands, which are public lands in the domain of the State. The Crees have exclusive trapping rights there (except in the southern zone), as well as certain non-exclusive hunting and fishing rights. Category III Lands include all of the lands within the territory covered by the James-Bay-Northern-Québec Agreement (JBNQA) that are located south of the 55th parallel and are not included in other land categories, notably Category I and II Lands for which Crees have exclusive hunting, fishing, and trapping rights.

20.3.2 Communities and First nations

The Eeyou Istchee James Bay Territory is made up of 16 communities inhabited by some 15,000 Crees and 15,000 Jamésiens. From north to south, the region encompasses nine Cree communities. Waswanipi is the closest Cree community to the Project site, about 300 km north-east. Crees have been living within the territory for about 5,000 years.

With an area of about 275,000 km^2, the municipality is the most widespread in Québec. The region's distinctive social, cultural, and economic fabric is the result of the coexistence of two cultures and the intermingling of their lifestyles and traditions, the Crees and the Jamésiens.

The Jamésien population is mainly concentrated in the localities of Valcanton, Villebois and Radisson, and in the four enclosed towns of Chapais, Chibougamau, Matagami and Lebel-sur-Quévillon. Matagami, the closest municipality, is about 100 km north of the Project site. There is no permanent village or settlement in the vicinity of the Project and there is also no public or private permanent infrastructure. The closest municipality to the south is more than 30 km away.

Further south, the Abitibiwinni First Nation of Pikogan is located about 75 km south of the mine site by road. This community is also present on the territory, carrying out hunting, fishing and trapping activities.

20.3.3 Territorial key characteristics

As part of the Canadian shield, the area is marked by its northern location, between the 49th and the 55th parallels, and by its magnitude, covering almost a quarter of the Province. The site is far from any urban area and population density is about 1 individual/100 km^2.

The weather is harsh, with long and cold winters, with temperature of about -15°C, and a short summer growing season, with temperature of 16°C. Precipitations are abundant.

Lakes and rivers are numerous and sometimes very large. Several streams flow toward the west side of the Property, reaching the Harricana River, which flows north to James Bay. Wetlands and forests are everywhere, the taiga and the tundra to the north and the boreal forest to the south and are mostly composed of coniferous trees. On the property, balsam fir, white birch, poplar, black spruce, jack pine and tamarack are abundant, as well as wetlands.

In such area, natural resources are abundant, with numerous fish, bird, and mammal species. Beaver is particularly abundant. The Harricana River flows about seven kilometres west of the Property. The Quebec government is planning to create an aquatic reserve to protect the Harricana River and a 200 m corridor on both riverbanks.

Considering the richness of the natural resources and the presence of First Nations, numerous protected areas have been set up by the government authorities. There are some 290 protected areas of various status and size, for up to 80,000 km^2. None of these protected areas are in the limits of the Property, the closest being more than 8 km away.

Much further north, the territory is also host to the hydroelectric complex of La Grande. Developed over many decades from the 1970s, La Grande is still today one of the largest man-made water reservoir complexes in the world.

Besides its energy potential, the area's economy is mainly oriented toward the exploitation of the natural resources, namely forestry and mining. Fishing, hunting, and trapping remain key activities, attracting tourism to outfitter, while being an integrated part of the culture and identity of the First Nations. For instance, Crees are well part of the economy and hold numerous compagnies in various domains such as construction and transportation.

20.3.4 Consultation

As the Project is still in its early stage, no consultation has been undertaken. Through the process of a CofA modification, the MELCCFP will be invited - along with the Pikogan First Nation community who have some access to the area of the Project - to give their opinions on the Project.

20.3.5 Agreements

So far, there have been no negotiations nor agreements between Abcourt and any group, organization, or First Nations regarding the Project.

20.4 Environmental and social issues

20.4.1 Context and approach

Impacts on any components are triggered by a source of impacts, which could be project infrastructures, works or activities. Impacts are typically observed during the construction, the operation, or the closure phase. Impacts are determined for each biophysical and social component, through the analysis of the technical characteristics of the Project, the knowledge of the surrounding environment and experience from similar projects.

The general approach for the identification of impacts is to analyze the interactions between the valued environmental and social components and the sources of impacts. Once the interrelations are identified, the significance of the impact is evaluated through a series of criteria such as the geographic extent of the impact, its magnitude, and its duration. Mitigation measures are then applied to lessen the identified impact. Finally, compensation measures can also be identified, if needed.

Potential impacts are typically driven by a few main elements, as follows:

- The location of the main infrastructures and the footprint of the Project in the surrounding environment.
- The choice of the design criteria regarding activities such as the mine planning,
- The management of waste and water, production schedule, setting the tone for the magnitude and the duration of the influence that the infrastructures will exercise on the surrounding environment.
- The construction of the Project infrastructures; including many works such as clearcutting, blasting, excavating, building, etc., that will change the landscape.
- The operation of the Project infrastructures; including mining, ore processing, handling of waste and tailings, water management and all the inputs and outputs required for the operation and produced throughout the operation.

20.4.2 Potential impacts and mitigation measures

Considering the key features of the Project and the setting of the receiving environment, potential impacts of the project will be influenced by a few features that will help to avoid and minimize the overall impacts, as follows:

- The underground mine and its infrastructures are already in place;
- Ore processing will be performed on site, with the processing plant and most of the infrastructures also in place; and
- The mining site is located away from any municipality or public/private infrastructure.

With the implementation of mitigation measures in the course of the Project, these features will contribute to avoiding and to minimizing the overall impacts of the Project on the environmental and social components.

20.4.2.1 Physical components

The key issue with the physical components will be to limit and control impacts of the infrastructures on the soil, water and air components, in order to avoid contamination of the environment.

20.4.2.2 Biological components

The key issue with the biological components will be to limit the effects of the Project footprint on the natural habitats.

20.4.2.3 Social components

The key issue with the social components will be to control detrimental effects of any nuisance on the land occupation and resources uses by the communities on the territory.

20.5 Mine closure requirements

The Mining Act (CQLR, c. M-13.1) and the Regulation concerning mineral substances other than petroleum, natural gas and brine (CQLR, c. M-13.1, r. 2) requires companies to restore the land affected by their mining and mineral exploration activities (Section 232.1). The Mining Act requires that a rehabilitation and restoration plan (also called closure plan) and financial guarantee covering the cost of restoration work are provided by corporations. The plan must be approved by the Ministère des Ressources naturelles et des Forêts (MRNF) prior to the commencement of mining activities in the case of exploration, and prior to the issuance of the mining lease for mining operations.

A mining lease is not required as the site is already covered by mining leases.

20.5.1 Requirements

A rehabilitation and restoration plan must be prepared according to the *Guide de préparation du plan de réaménagement et de restauration des sites miniers au Québec* (MERN, 2022). The plan must be reviewed every five years. Significant changes to the Project may also trigger the need to update the plan, as requested by the MRNF. A financial guarantee must be provided to the MRNF that covers the total estimated costs (100%) of the restoration plan.

The EQA requires that for industrial and commercial activities listed in Schedule III of the Regulation for the Protection and Rehabilitation of Lands ("RPRL", Q-2, r.37), such as gold and silver ore mining or processing (NAICS No.21222), an Environmental Site Assessment (Phases I and II) should be performed within 6 months following the cessation of activities. If contaminants exceeding the limits stated in the regulation are measured, a rehabilitation plan would have to be submitted to the MELCCFP. Following its approval, the rehabilitation works would have to be conducted in order to comply with the plan and for future site utilization.

In 2018, an updated version of the rehabilitation and restoration plan for the Sleeping Giant mine was filed to the MRNF by Abcourt. Subsequently, Abcourt received comments from the MRNF on the closure plan. Responses were submitted to the MRNF in February 2023 in which a revised cost estimate for site closure was included. Abcourt is currently waiting for feedback from the MRNF.

As the proposed activities related to the Project would be similar to the activities conducted in the last few years, the answers presented to the MRNF will not differ much from those submitted, especially for the TSF. Nevertheless, if the MRNF considers that significant changes are related to the Project, it can trigger the need for an updated closure plan.

20.5.2 Closure concept

Past geochemical results showed that the PN/PA ratio related to the contents of the TSF is 0.9 and therefore has a potential for acid generation. The characteristics of the mineralized zones that will be mined are expected to have sulfur in the form of sulphide greater than 0.04%, a PN/PA ratio around 1 and a PNN that can exceed 20, or even reach 50.

The restoration concept approved by the MRNF in the previous versions of the closure plan is maintained. Agricultural lime used as an amendment will be mixed with the first 30 cm of mine tailings for the restoration of the TSF. It is anticipated that the geochemical characteristics of the anticipated ore feed, and therefore of future mine tailings, will be similar to the former mineralized zones previously mined. Additional tests will be required to assess the dosage required for the anticipated agricultural lime used as an amendment to meet the current requirements of the *Guide de caractérisation des résidus miniers et du minerai* (MELCC, 2020).

In addition to the environmental monitoring program, a physical stability monitoring program will be implemented following the site closure to evaluate the integrity of the existing structures. Visual inspections of the dikes (including the TSF) and waste rock piles will be conducted in order to observe anomalies that could jeopardize the stability. Water discharge points will also be monitored for signs of erosion. The monitoring programs will be conducted during the post-mining and post-restoration periods. The post-mining period is expected to be two years.

Following revegetation of the site, agronomic monitoring will be undertaken. Annual visits consisting of visual observations and sample collection will be conducted. The samples will be analyzed in order to observe if deficiencies are present. Corrective actions would be taken if necessary. Organic matter will be added after each year for the first 3 years if required.

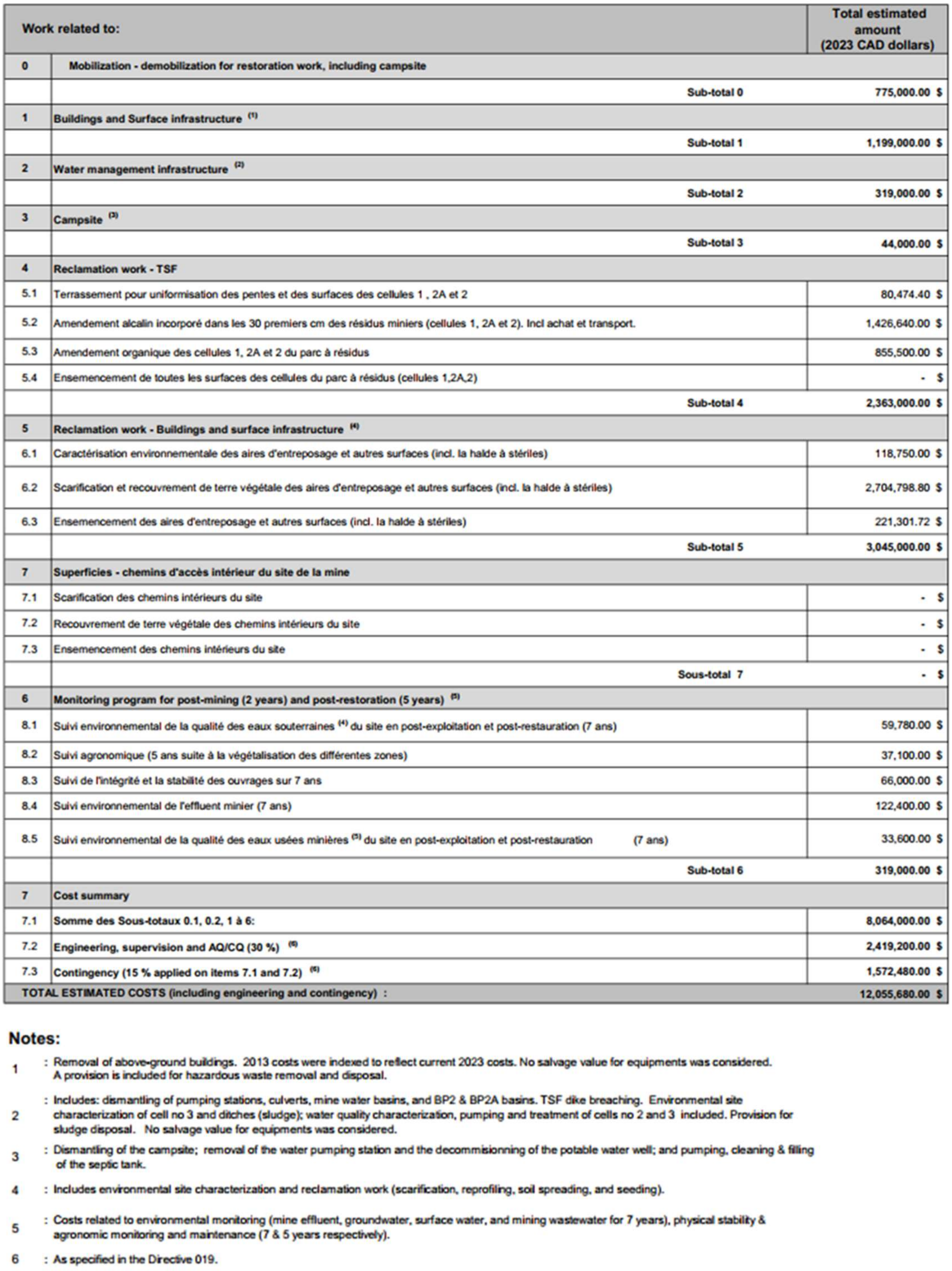
Environmental monitoring, which includes the mine effluent (discharge of the BP-2A basin) and the groundwater quality, is planned to be conducted for a period of five years during the post-restoration period. After this period, the monitoring could be abandoned by Abcourt under certain conditions, such as the results of the monitoring programs fulfill the requirements of Directive 019.

The rehabilitation and closure costs have been estimated at $12.1 million (M), including the closure bond held by the MRNF presently at $5.4 M and $6.7 M remaining in capital expenditure. As stated previously, Abcourt is currently waiting for the feedback of the MRNF for the reviewed closure cost estimate.

The cost estimate provided takes into consideration the dismantlement of surface structures, new accommodation facilities, water management infrastructure and the restoration of the TSF. For this estimate, it was assumed that the restoration work will be carried out by a third party, as per the MRNF guide. A salvage value was not considered for the resale of equipment or materials.

Details for the closure costs of the Property related to the total financial guarantee to be provided to the MRNF are summarized in Table 20.1.

Table 20.1 – Cost Estimate for Mine Closure Work for the Sleeping Giant Mine



Work related to:		Total estimated amount (2023 CAD dollars)
0	**Mobilization - demobilization for restoration work, including campsite**	
	Sub-total 0	**775,000.00 $**
1	**Buildings and Surface infrastructure (1)**	
	Sub-total 1	**1,199,000.00 $**
2	**Water management infrastructure (2)**	
	Sub-total 2	**319,000.00 $**
3	**Campsite (3)**	
	Sub-total 3	**44,000.00 $**
4	**Reclamation work - TSF**	
5.1	Terrassement pour uniformisation des pentes et des surfaces des cellules 1 , 2A et 2	80,474.40 $
5.2	Amendement alcalin incorporé dans les 30 premiers cm des résidus miniers (cellules 1, 2A et 2). Incl achat et transport.	1,426,640.00 $
5.3	Amendement organique des cellules 1, 2A et 2 du parc à résidus	855,500.00 $
5.4	Ensemencement de toutes les surfaces des cellules du parc à résidus (cellules 1,2A,2)	- $
	Sub-total 4	**2,363,000.00 $**
5	**Reclamation work - Buildings and surface infrastructure (4)**	
6.1	Caractérisation environnementale des aires d'entreposage et autres surfaces (incl. la halde à stériles)	118,750.00 $
6.2	Scarification et recouvrement de terre végétale des aires d'entreposage et autres surfaces (incl. la halde à stériles)	2,704,798.80 $
6.3	Ensemencement des aires d'entreposage et autres surfaces (incl. la halde à stériles)	221,301.72 $
	Sub-total 5	**3,045,000.00 $**
7	**Superficies - chemins d'accès intérieur du site de la mine**	
7.1	Scarification des chemins intérieurs du site	- $
7.2	Recouvrement de terre végétale des chemins intérieurs du site	- $
7.3	Ensemencement des chemins intérieurs du site	- $
	Sous-total 7	**- $**
6	**Monitoring program for post-mining (2 years) and post-restoration (5 years) (5)**	
8.1	Suivi environnemental de la qualité des eaux souterraines (4) du site en post-exploitation et post-restauration (7 ans)	59,780.00 $
8.2	Suivi agronomique (5 ans suite à la végétalisation des différentes zones)	37,100.00 $
8.3	Suivi de l'intégrité et la stabilité des ouvrages sur 7 ans	66,000.00 $
8.4	Suivi environnemental de l'effluent minier (7 ans)	122,400.00 $
8.5	Suivi environnemental de la qualité des eaux usées minières (5) du site en post-exploitation et post-restauration (7 ans)	33,600.00 $
	Sub-total 6	**319,000.00 $**
7	**Cost summary**	
7.1	**Somme des Sous-totaux 0.1, 0.2, 1 à 6:**	**8,064,000.00 $**
7.2	**Engineering, supervision and AQ/CQ (30 %) (6)**	**2,419,200.00 $**
7.3	**Contingency (15 % applied on items 7.1 and 7.2) (6)**	**1,572,480.00 $**
TOTAL ESTIMATED COSTS (including engineering and contingency) :		**12,055,680.00 $**

Notes:

1 : Removal of above-ground buildings. 2013 costs were indexed to reflect current 2023 costs. No salvage value for equipments was considered. A provision is included for hazardous waste removal and disposal.

2 : Includes: dismantling of pumping stations, culverts, mine water basins, and BP2 & BP2A basins. TSF dike breaching. Environmental site characterization of cell no 3 and ditches (sludge); water quality characterization, pumping and treatment of cells no 2 and 3 included. Provision for sludge disposal. No salvage value for equipments was considered.

3 : Dismantling of the campsite; removal of the water pumping station and the decommisionning of the potable water well; and pumping, cleaning & filling of the septic tank.

4 : Includes environmental site characterization and reclamation work (scarification, reprofiling, soil spreading, and seeding).

5 : Costs related to environmental monitoring (mine effluent, groundwater, surface water, and mining wastewater for 7 years), physical stability & agronomic monitoring and maintenance (7 & 5 years respectively).

6 : As specified in the Directive 019.

In addition, the rehabilitation and closure work must begin within three years after the operations cease, unless the Minister may exceptionally authorize an extension (c. M-13.1, s.232.7.1).

21. CAPITAL AND OPERATING COSTS

The capital and operating cost estimates presented in the 2023 PEA are based on the resumption of underground operations at a production rate of 350 tonnes per day following the re-commissioning of surface and underground infrastructure including the addition of accommodation and catering modules for on-site workers.

All capital and operating cost estimates cited in the Report are referenced in Canadian dollars ("CAD" or "$").

21.1 Capital Costs

21.1.1 Summary

The cumulative life of mine (LOM) capital expenditure ("CAPEX") costs, including initial investment capital cost (pre-production period) and the sustaining cost are estimated at $97.8 million ($M).

This excludes the production revenue of $18.7 M expected during the pre-production period and the closure bond currently deposited with the Ministère des Ressources naturelles et des Forêts ("MRNF") of $5.4 M.

The total pre-production capital costs for Sleeping Giant project are estimated at $60.6M. These estimates include the addition of certain contingencies and indirect costs. The total sustaining costs for the remaining life of the Sleeping Giant project life (following the preproduction period) is estimated at $37.2M. This includes ongoing capital development, on-going surface and underground infrastructure and site reclamation and closure cost.

Total pre-production and sustaining capital costs are summarized in Table 21.1 and Figure 21.1.

Table 21.1 – Project capital cost summary

Capital Expenses[1] (M CAD)	Total Capital Costs ($M)
Pre-production operating expenses	27,1
Mine development	28,7
Infrastructures	6,0
Equipment reconditioning	6,7
Electrical circuit	6,4
Tailing facilities	10,9
Closure and reclamation	12.1
Total capital cost	**97.8**
Pre-production revenue NSR[2]	(18.7)
MRNF closure bond deposit[3]	(5.4)
Total net capital cost	**73.8**

1. The total can vary due to rounding.
2. The Corporation expects revenues during the pre-production period.
3. Net amount including the closure bond currently deposited with the MRNF for 5.4 M$.

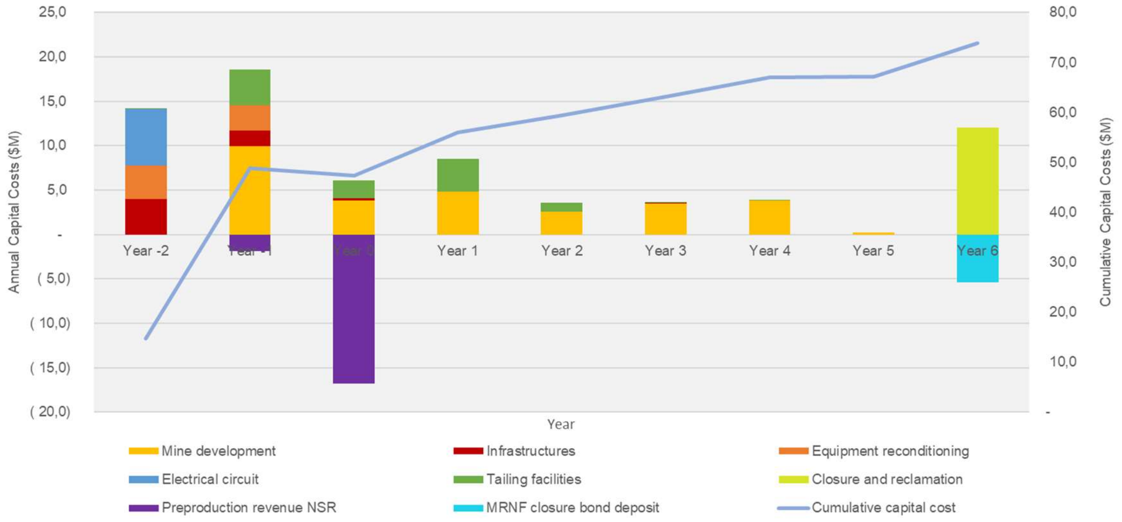


Figure 21.1 – Overall Sleeping Giant capital cost profile

21.1.2 Scope and Structure of Capital Cost Estimate

The cost schedule estimated for the LOM was divided into two timeframes: pre-production and steady state production. Pre-production capital costs include investment costs incurred during the first 18 months of the project.

The overall capital cost estimate developed in this Preliminary Economic Assessment Study has an accuracy range of between -30% and +30%. The capital cost estimate for this Study forms the basis for the approval of further development of the Project by means of a Feasibility Study. The various sections have been developed at different levels of engineering definition. Some sections are as low as 1%, whereas others are up to 30%.

The capital cost estimate abides by the following criteria:

- It reflects general accepted practices in the cost engineering profession;
- Its labour costs are based on current Québec (“QC”) rates;
- The capital costs are expressed in constant Q1 2023 Canadian dollars.
- All costs in US dollars were converted to Canadian using an exchange rate of 0.77 USD/CAD.

21.1.2.1 Estimate Responsibilities

Capital costs were estimated based on estimates provided by various mining industry consultants.

Table 21.2 summarizes the breakdown of responsibilities for capital costs evaluation:

Table 21.2 – CAPEX estimate responsibilities per activities

Work breakdown description	Responsible entity
Pre-production capitalised operating expenses	InnovExplo, Soutex, Abcourt
Process plant equipment	WSP
Process plant infrastructure	WSP
Surface infrastructure	WSP, Abcourt
Underground infrastructure	InnovExplo, WSP
Underground equipment	InnovExplo, Abcourt
Tailings, waste and water management	Englobe
Closure and Reclamation	Englobe

21.1.2.2 Exclusions

The following items are excluded from the capital cost estimate:

- Permitting cost;
- Licensing and financing cost;
- Project development costs incurred to date, including studies and early works;
- Foreign exchange variations;
- Taxes (included in the financial model);
- GST/QST tax;
- Project financing and interest charges;
- Price/cost escalation during construction;
- Import duties and custom fees;
- Pilot plant and other testwork;
- Operating costs;
- Sunk cost;
- Exploration activities;
- Severance cost for employees at the cessation of operations and;
- Any additional costs (but can partly be absorbed in contingency allowance).

21.1.3 Pre-production Capital Costs

The initial capital costs cover the capital expenses incurred during the pre-production period of the project. This period includes the following activities:

- Process plant infrastructures and equipements reconditionning
- Restoration of various surface infrastructures as required
- Installation of accommodation and catering facilities for on-site workers, including drinking water and wastewater services.
- Restoration of underground infrastructure to operational condition on the various levels, including the various networks.
- Rehabilitation of the escapeway and the various development access drift on the different levels.
- Refurbishment of underground equipment (Abcourt currently owns all the equipment required for the project).

The Project pre-production capital cost summary is outlined in Table 21.3. The capital cost breakdown descriptions are outlined in the following sections.

Table 21.3 – Project Pre-production capital cost summary

Area cost description	Pre-production Capital Costs ($M)	CAPEX (%)
Pre-production capitalised operating expenses	27,1	45%
Mine development	10,5	17%
Infrastructures	5,8	10%
Equipment reconditioning	6,7	11%
Electrical circuit	6,4	11%
Tailing facilities	4,1	7%
Closure and reclamation	0,0	0,0%
Total	**60.6**	**100%**

21.1.4 Sustaining Capital Cost

The sustaining capital cost represents the capital cost required to maintain the project after the pre-production period, broken down according to needs throughout the project's mine life.

For the Sleeping Giant project, the main sustaining capital costs are as follows:

- Underground development and rehabilitation
- On-going surface and underground infrastructure reconditioning
- Cost related to tailings facility management
- Site reclamation and closure cost

The Project Sustaining capital cost summary is outlined in Table 21.4.

Table 21.4 – Project sustaining capital cost summary

Area cost description	Sustaining Capital Costs ($M)	CAPEX (%)
Mine development	18,2	49%
Infrastructures	0,2	1%
Equipment reconditioning	0,0	0%
Electrical circuit	0,0	0%
Tailing facilities	6,7	18%
Closure and reclamation	12.1	32%
Total	**37.2**	**100%**

21.1.5 Capital cost breakdown

The capital cost breakdown for the Sleeping Giant project for the pre-production and sustaining capital cost are outlined in the following sections.

21.1.5.1 Pre-production capitalised operating expenses

Pre-production costs shown in Table 21.5 represent the direct costs incurred in extracting ore prior to reaching full production. This mainly includes:

- The development carried out to access and extract ore in the short term (Operating expenses ("OPEX") development)
- Direct production cost (drilling, blasting, mucking)
- Processing cost
- Related services to support the production

Table 21.5 – Pre-production capitalised operating expenses

Area cost description	Pre-production capital cost ($M)	Sustaining capital cost ($M)	Total capital cost ($M)	CAPEX (%)
Capitalised operating expenses	27,1	0.0	27,1	100%

21.1.5.2 Infrastructures

It was considered that the various infrastructures already in place on the site would be adequate for the project, following any necessary reconditioning work. The only new addition to the infrastructure will be the accommodation and catering modules.

WSP were commissioned to evaluate and estimate the reconditioning of the following infrastructures:

- Surface Infrastructures, including the process plant
- The accommodation and catering modules
- Underground loading facilities

InnovExplo has provided cost estimates for the reconditioning of the various underground level infrastructures (garage, explosives chamber, dining room, etc.).

Englobe as provided the cost estimate for the drinking water and Wastewater Services.

Table 21.6 provides the infrastructures cost breakdown.

Table 21.6 – Infrastructures cost

Area cost description	Pre-production capital cost ($M)	Sustaining capital cost ($M)	Total capital cost ($M)	CAPEX (%)
Surface infrastructures	0.9	0.0	0.9	15%
Accommodation and catering facilities	2.4	0.0	2.4	39%
Drinking water and wastewater services	0.8	0.0	0.8	14%
Underground infrastructures	1.4	0.2	1.6	26%
Contingency	0.5	0.0	0.3	8%
Total	**5.8**	**0.0**	**3.8**	**100%**

21.1.5.3 Mine development

Considering that a significant majority of production will be exploited from existing levels, a significant portion of the costs is associated with the rehabilitation of the different existing development.

The various orepass and wastepass network are fully operational and will be used to transfer material to the shaft.

InnovExplo has provided the CAPEX development cost estimates shown in Table 21.7.

Table 21.7 – Mine development

Area cost description	Pre-production capital cost ($M)	Sustaining capital cost ($M)	Total capital cost ($M)	CAPEX (%)
Lateral development	5.9	14.5	20.4	71%
Lateral development rehabilitation	2.4	3.7	6.1	21%
Vertical development	0.6	0.0	0.6	2%
Escapeway rehabilitation	0.5	0.0	0.5	2%
Shaft rehabilitation	1.1	0.0	1.1	4%
Total	**10.5**	**18.2**	**28.7**	**100%**

21.1.5.4 Electrical network

WSP was responsible for estimating the costs associated with updating the electrical network on the site. Table 21.8 provides the breakdown related to the electrical network.

The main costs are as follows:

- Upgrading of the site main electric substation with ground fault protection,
- Modification of the Hydro-Québec 25 kV overhead line in order to connect the new camp modules;
- Electrification of the on site camp;
- Reconditioning of the electrical substations on the various underground levels.

Table 21.8 – Electrical network

Area cost description	Pre-production capital cost ($M)	Sustaining capital cost ($M)	Total capital cost ($M)	CAPEX (%)
Surface main substation	1.8	0.0	1.8	29%
Process Plant upgrades	0.7	0.0	0.7	11%
Emergency generator	0.2	0.0	0.2	3%
Accommodation facilities electrification	0.4	0.0	0.4	7%
Underground substation	1.7	0.0	1.7	27%
Indirect Cost (15%)	0.7	0.0	0.7	12%
Contingency (15%)	0.7	0.0	0.7	12%
Total	**6.4**	**0.0**	**6.4**	100%

21.1.5.5 Tailings facilities

It is expected that existing deposition cells will be used. Dyke raising and dredging works are planned during the project.

Englobe has provided the cost estimates shown in Table 21.9 for the tailing facilities.

Table 21.9 – Tailings facilities

Area cost description	Pre-production capital cost ($M)	Sustaining capital cost ($M)	Total capital cost ($M)	CAPEX (%)
Engineering, Feasibility Study & Strategic Restauration Plan Update	0.5	0.0	0.5	5%
Mobilisation, Traffic Management and Monitoring of works	0.7	0.0	0.7	6%
Construction work on tailings facility	7.1	0.0	7.1	67%
Dredging works	2.5	0.0	2.5	23%
Total	**10.9**	**0.0**	**10.9**	**100%**

21.1.5.6 Closure cost and mine reclamation

Englobe has provided the cost estimates for the closure and reclamation.

Table 21.10 provide the breakdown of the closure cost and mine reclamation.

Table 21.10 – Closure cost and mine reclamation

Area cost description	Pre-production capital cost ($M)	Sustaining capital cost ($M)	Total capital cost ($M)	CAPEX (%)
Mobilization and Demobilization	0.0	0.9	0.9	7%
Dismantling of structures on site	0.0	1.8	1.8	15%
Reclamation Works TSF, Buildings and Surface Infrastructures	0.0	6.2	6.2	52%
Monitoring program for post-mining (2 years) and post-restoration (5 years)	0.0	0.4	0.4	3%
Engineering, supervision and AQ/CQ (30 %)	0.0	2.8	2.8	23%
Total	**0.0**	**12.1**	**12.1**	**100%**

21.2 Operating Costs

21.2.1 Summary

The average operating cost over the six-year mine life is estimated to be $320.67/t mined or $1260/oz (CAD).

The operating unit costs were calculated over the total mineralized material mined from development and from production, excluding the tonnage during preproduction. Table 1.6 and Table 21.12, provides the breakdown of the projected operating costs and the annual cost profile for the Sleeping Giant Project.

Table 21.11 – Sleeping Giant operating cost summary

Area cost description	LOM Operating Cost ($M)	LOM Average Operating Cost per year ($M)	LOM Average Operating Unit Cost ($/t)	LOM Average Operating Unit Cost ($/oz)	OPEX (%)
Definition Drilling	10.8	1.9	15.86	62.3	5%
Stopes Preparation	19.4	3.4	28.45	111.8	9%
Mine Production	67.9	11.8	99.79	392.0	31%
Mine Services	39.7	6.9	58.32	229.1	18%
Maintenance	16.2	2.8	23.81	93.5	7%
Technical Services	19.2	3.3	28.19	110.8	15%

Area cost description	LOM Operating Cost ($M)	LOM Average Operating Cost per year ($M)	LOM Average Operating Unit Cost ($/t)	LOM Average Operating Unit Cost ($/oz)	OPEX (%)
Mill Processing	32.3	5.6	47.17	186.5	1%
Environment	1.2	0.2	1.76	6.9	5%
General and Administration (“G&A”)	11.6	2.0	17.01	66.8	15%
Total	**218.3**	**37.9**	**320.67**	**1259.7**	**100%**

Table 21.12 – Sleeping Giant annual operating cost profile

Area cost description	2025	2026	2027	2028	2029	2030
Definition Drilling	1.7	2.3	2.3	2.3	2.1	0.0
Stopes Preparation	4.1	3.6	4.0	3.7	3.0	0.9
Mine Production	9.2	12.1	11.3	11.9	13.4	10.1
Mine Services	5.1	7.1	7.1	7.2	7.1	6.0
Maintenance	2.2	2.9	2.9	2.9	2.9	2.4
Technical Services	2.6	3.5	3.5	3.5	3.5	2.6
Mill Processing	4.5	5.8	6.0	6.0	6.0	4.0
Environment	0.2	0.2	0.2	0.2	0.2	0.2
General and Administration (“G&A”)	1.5	2.0	2.0	2.0	2.0	2.0
Total	**31.1**	**39.5**	**39.4**	**39.6**	**40.4**	**28.3**

21.2.2 Basis of operating cost estimate

The operating cost estimate was based on Q1 2023 assumptions. The estimate has been deemed to be of an accuracy within alignment with a PEA level of study. All operating cost estimates are in CAD. Mining, process, technical services and environment are generally itemized in detail, however, General and Administration (“G&A”) items are calculated estimates based on extrapolation of historical costs by previous operators. Many elements of the operating cost estimates are based on budget estimates, and salaries and allowances are based on the latest Mine Abcourt collective agreement salary scales.

The operating cost estimate is based on the mine schedule indicative tonnage per time period that was produced by InnovExplo on May. See Chapter 16 for more details related to the mine plan.

Assumptions and Exclusions

The following items were assumed:

- Abcourt owns all the material and equipment required for the project, which will be reconditioned and reused;
- Labour rates include, where applicable, performance bonuses, fringe benefits, transport allowances and on-site accommodation costs;
- Freight estimates are based on vendor supplied freight quotations or in-house data. Freight for reagents is included in the price of those commodities. Freight for steel consumables is included in the price of that material. Freight for spare parts is calculated as a percentage of equipment cost expected to be used annually;
- No contingency is assumed;
- No cost escalation (or de-escalation) is assumed;
- No costs relating to certain agreements with third parties.

The following items were specifically excluded from the operating cost estimate:

- Cost of financing and interest;
- Pre start-up operations and maintenance training;
- Transport and handling of Doré from the plant (included in the financial analysis).

Estimate Responsibilities

The overall operating cost estimate combined inputs from a number of sources including InnovExplo, WSP, Englobe and Abcourt as summarized in Table 21.13.

Table 21.13 – OPEX estimate responsibilities

Work breakdown description	Responsible entity
Mining cost	InnovExplo, WSP, Abcourt
Mill processing	Soutex, Abcourt
Tailings, waste and water management	Englobe
Technical services	InnovExplo
Environment	Englobe
General and Administration	InnovExplo

General Unit Rates

General rates used in the operating cost estimate are summarized in Table 21.14.

Table 21.14 – General rate and unit cost assumptions

Parameter	Unit	Value
Average Daily LOM Tonnage	tpd	350
Years of operations	Years	5.8
LOM production	oz (000)	178.3
LOM gold grade	Au g/t	8.1
Power at site	$/kWh	0.063
Propane	$/litre	0.753

21.2.3 Underground Mining Cost and Surface services

InnovExplo with Abcourt provided estimates for all underground mine operating costs including surface services. The total mine operating cost is $154.0 M for the Sleeping Giant Project. The unit cost is $226.24 $/t mined.

Mining operating costs constitute in their majority of wages, electric power, consumables and equipment maintenance. All lateral and vertical development in ore material and waste material that procured immediate access to production stoping has been allocated to operating cost and considered in the stope preparation cost area.

Table 21.15 summarizes the underground mining operating costs for Sleeping Giant and provides a breakdown per item.

Table 21.15 – Underground mining and surface services operating cost

Activity	Sub-activity	Total LOM cost ($M)	Average LOM unit cost ($/t)	OPEX (%)
Definition Drilling		10.8	15.86	7%
Stope preparation		19.4	28.45	13%
Production	Long hole mining method	4.9	7.21	3%
	Room and Pillars mining method	17.4	25.63	11%
	Shrinkage mining method	21.2	31.17	14%
	Mucking	18.5	27.20	12%
	Slushing	5.8	8.59	4%
Mines services		28.1	41.32	18%
Maintenance	Mechanics services	10.7	15.68	7%
	Electric services	5.5	8.13	4%
	Surface services	11.6	17.00	8%
Total		**154.0**	**226.24**	**100%**

21.2.4 Mill processing

Soutex provided estimates for the mill processing operating costs. The total operating cost is $32.3 M for the Sleeping Giant Project. The unit cost is $47.17 $/t mined.

Table 21.15 summarizes the mill processing costs for Sleeping Giant and provides a breakdown per item.

Table 21.16 – Mill processing operating cost

Activity	Total LOM cost ($M)	Average LOM unit cost ($/t)	OPEX (%)
Labors	18.0	26.21	56%
Energy	2.6	3.77	8%
Maintenance and wear parts	4.4	6.41	14%
Grinding media	1.4	2.08	4%
Reagents and consumables	5.2	7.58	16%
Laboratory	0.8	1.12	2%
Total	**32.3**	**47.17**	**100%**

21.2.5 Technical services

InnovExplo and Abcourt provided estimates for the technical services operating costs. The total operating cost is $19.2 M for the Sleeping Giant Project. The unit cost is $28.19 $/t mined.

Table 21.17 summarizes the technical services costs for Sleeping Giant and provides a breakdown per item.

Table 21.17 – Technical services operating cost

Activity	Total LOM cost ($M)	Average LOM unit cost ($/t)	OPEX (%)
Engineering services	7.2	10.51	37%
Geology services	12.0	17.68	63%
Total	**19.2**	**28.19**	**100%**

21.2.6 Environment

Englobe provided estimates for the environment operating costs. The total operating cost is $1.2 M for the Sleeping Giant Project. The unit cost is $1.76 $/t mined.

Table 21.18 summarizes the environment costs for Sleeping Giant and provides a breakdown per item.

Table 21.18 – Environment operating cost

Activity	Total LOM cost ($M)	Average LOM unit cost ($/t)	OPEX (%)
Labors	0.7	1.08	62%
Ferric sulfate	0.1	0.16	9%
Energy	0.1	0.09	5%
Water assays	0.3	0.43	25%
Total	**1.2**	**1.76**	**100%**

21.2.7 General and Administration

InnovExplo provided estimates for the G&A operating costs. The total operating cost is $11.6 M for the Sleeping Giant Project. The unit cost is $17.01 $/t mined.

G&A operating costs, excluding labour costs, have been estimated on the basis of updated historical data.

These estimates include:

- General insurance
- Taxes
- Permit and mining lease fees
- Association and convention fees
- Concierge services
- Professional fees
- IT
- Recruitment costs
- Stationery and office supplies
- Miscellaneous supplies

Table 21.19 summarizes the environment costs for Sleeping Giant and provides a breakdown per item.

Table 21.19 – General and Administration operating cost

Activity	Sub-activity	Total LOM cost ($M)	Average LOM unit cost ($/t)	OPEX (%)
Labors		7.8	11.41	67%
Consultancy, fees, miscellaneous expenses and materials	Management	2.9	4,19	25%
	Procurement and warehouses	0.1	0,09	1%
	Human resources	0.4	0,56	3%
	Health and safety	0.5	0,76	4%
Total		**11.6**	**17.01**	**100%**

21.3 Site personnel

A total facility workforce of 111 employees is estimated for the Sleeping Giant project. A summary of the workforce distribution by area is shown in Table 21.20.

Table 21.20 – Summary of total workforce distribution by area

Facility area	Role	Total
Underground Mine	Staff and Supervision	13
	Operations	18
	Maintenance	20
Processing	Staff and Supervision	3
	Operations	20
Technical services	Engineering	9
	Geology	15
General and Administration	Management	2
	Administration	1
	Human resources and Community relations	8
	Health and Safety	1
	Environment	1

22. ECONOMIC ANALYSIS

22.1 Overview

The economic/financial evaluation of the Project was carried out using the discounted cash flow approach on a pre-tax and after-tax basis. The gold price was derived from a consensus based on long-term average commodity prices in USD, and expenditure estimates in the currency in which they were made. An equivalent exchange rate of 0.77 USD to 1.00 CAD was used to convert the market price of raw materials into CAD. No provision has been made for the effect of inflation. Current Canadian corporate tax regulations have been considered for the assessment of federal mining tax liabilities, and the most recent provincial tax regulations have been used to access the Québec mining tax liabilities.

The internal rate of return (IRR) was calculated without considering any financing costs. Net present value (NPV) was determined by the pre-tax and after-tax cash flow generated by the project using a 5% discount rate. Payback periods following the full production are based on the undiscounted annual cashflow of the project. Sensitivity analysis on the pre-tax and after tax for both NPV and IRR have been performed to assess the impact of variations on the gold price, capital cost, operating cost and the revenues (variation of the head grade).

The economic analysis presented in this chapter are based on many forward-looking information and the results are subject to known and unknown risks, uncertainties and many other factors which could materially alter the results obtained in relation to those presented in this report.

22.2 Assumptions

- The cash flow, which served as the basis for the economic analysis, was generated from the mining plan presented in Chapter 16;
- The base-case gold price for the economic evaluation of the project is 1,800 USD/oz. The gold price was set according to a consensus of experts discussions based on a long-term average market price. No annual indexation of price was considered. Metal prices can be highly volatile and can be influenced by multiple factors;
- The Project was considered in commercial production following three months of full production at 350 tpd and a positive cash flow, which corresponds to a period of 18 months after the start of pre-production work;
- All costs are in constant CAD of Q1 2023 without any consideration of inflation or cost escalation;
- The exchange rate assumed between the USD and CAD dollars was 0.77 (CAD:USD exchange rate of 1.30);
- Project revenue is based on sale of gold doré in the international marketplace;
- Revenues generated during the pre-production period were capitalized;
- Reclamation and closure costs include a closure bond of $5.4 M held by the MNRF.

The financial analysis was performed on both pre-tax basis and after-tax basis. Table 22.1 shows the main parameters.

Table 22.1 – Financial base parameters

Description	Unit	Value
Gold price	USD/oz	1,800
Exchange rate	USD:CAD	0.77
Discount rate	%	5
Mine life (including pre-production, reclamation and closure period)	Years	8
Total tonnage mined	t (000)	720
Gold grade	g/t	8.10
Process overall gold recovery	%	96.7
Operating mining cost	$/t mined	320.67
Royalties	% NSR	2
Pre-production capital cost[1]	$M	42.0
Sustaining capital cost[2]	$M	31.8

1. Includes capitalised revenues generated during pre-production period;
2. Includes a closure bond of $5.4 M held by the MRNF.

22.3 Royalties

Over the life of mine, based on the agreement in place, a 2% NSR royalty has been assumed for the Project. For the entire Project, this represents a total payable charge of $8.5 M based on the base-case gold price and Project assumptions.

22.4 Taxation

The taxation for the Project was compiled with the assistance of finance experts Raymond Chabot Grant Thornton.

22.4.1 Taxes Assumptions

Tax rates will remain unchanged from the latest rates, practically in force for the entire duration of the Project covered by this tax estimate.

The Project is located in Québec and is therefore subject to the following mining taxes and duties:

Corporate income tax

- The federal tax rate for 2022 and beyond is 15 %;
- The Quebec tax rate for 2022 and beyond is 11.5 %;
- Capital cost allowance is calculated at a rate of 25 % on a declining balance basis.

Mining taxes

Under the Mining Tax Act, an operator is required to pay mining duties for a fiscal year equal to the greater of its mining tax on annual profit and its minimum mining tax for the fiscal year. An operator's minimum mining tax for a fiscal year is calculated on the value of production at the pithead of all the mines it operates. An operator's annual profit for a fiscal year is determined by considering the total annual profits of each mine it operates, less certain expenses and allowances. For present purposes, a rate of 16% has been used to calculate the minimum tax, and only expenses related to machining, processing and transportation have been considered.

Tax treatment of capital expenditure

Capitalized operating costs have been treated as development costs for corporate income tax purposes and as pre-production development costs for mining income tax purposes.

Development costs have been treated as development costs for corporate income tax purposes and as post-production development costs for mining income tax purposes.

The tax calculations are underlined by the following key assumptions:

- Calculations are based on the tax regime at the time of the PEA. The calculations could be affected by future changes in tax laws;
- Tax attributes totaling $29.0 M have been considered;
- A 100% corporate entity owns the Project, and the after-tax analysis does not take into account any future changes in corporate structure;
- It assumes 100% equity funding and does not take into account interest or financing costs;
- The actual taxes payable will be affected by corporate activities, as well as current and future tax benefits.

22.5 Financial Analysis Summary

A discount rate of 5% was applied to the cashflow to derive the NPV for the projection a pre-tax basis and after-tax basis. The summary of the financial evaluation for the base case of the project is presented in Table 22.2 and Table 22.3.

Table 22.2 – Financial analysis summary (pre-tax)

Pre-Tax

Items	Unit	Value
Net Present Value (5%)	$M	77.5
Internal Rate of Return	%	41.1
Payback period	Years	2.1

Table 22.3 – Financial analysis summary (after-tax)

After-Tax

Items	Unit	Value
Net Present Value (5%)	$M	54.4
Internal Rate of Return	%	33.3
Payback period	Years	2.2

For the pre-tax base case, the financial model resulted in an IRR of 41.1% and a NPV of $77.5 M with a discount rate of 5%. The payback period following the start of the commercial production is 2.1 years. On an after-tax basis, the financial model resulted in an IRR of 33.2% and a NPV of $33.2 M with a discount rate of 5%. The payback period following the start of the commercial production is 2.2 years.

Table 22.4 shows the financial model summary of the discount cash flow for the Project.

Table 22.4 – Sleeping Giant financial model summary

	Unit	Year-2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Production Summary										
Total Tonnes Milled	kt		3,5	127,7	123,6	127,5	126,2	127,5	84,2	
Head Grade Au	g/t		7,25	6,82	8,37	9,63	7,31	7,68	9,20	
Total Gold Produced	koz		0,8	27,0	32,1	38,2	28,6	30,4	24,1	
Revenue										
Gross Revenue	$M		1,9	63,2	75,2	89,5	67,0	71,1	56,4	
Operating Expenditures										
Mining	$M			22,3	28,0	27,6	27,9	28,6	19,5	
Processing	$M			4,5	5,8	6,0	6,0	6,0	4,0	
General & Administration[1]	$M			4,3	5,7	5,7	5,7	5,7	4,8	
Operating cost	**$M**			31,1	39,5	39,4	39,6	40,4	28,3	
Royalty payments	$M			1,3	1,5	1,8	1,3	1,4	1,1	
Capital Expenditures										
Pre-production[2]	$M	14,7	34,1	(6,8)						
Sustaining	$M			5,4	8,5	3,5	3,6	3,9	0,2	0,0
Reclamation and Closure[3]	$M									6,7
Pre-Tax Cash Flow										
Pre-Tax Cash Flow	$M	(14,7)	(34,1)	15,3	25,4	44,6	22,3	25,3	26,7	(6,7)
Cumulative Pre-Tax Cash Flow	$M	(14,7)	(48,8)	(33,6)	(8,1)	36,5	58,7	84,0	110,7	104,1
Income Taxes										
Pre-Tax Cash Flow	$M					2,9	3,0	4,3	4,4	(4,5)
Cumulative Pre-Tax Cash Flow	**$M**		0,0	0,6	1,0	6,6	3,2	3,9	3,7	0,0
After-Tax Cash Flow										
After-Tax Cash Flow	$M	(14,7)	(34,1)	14,7	24,4	35,1	16,1	17,2	18,6	(2,2)
Cumulative After-Tax Cash Flow	**$M**	(14,7)	(48,8)	(34,2)	(9,7)	25,4	41,5	58,7	77,3	75,0

1. Includes Technical services and Environment areas;
2. Includes capitalised revenues generated during pre-production period;
3. Includes a closure bond of $5.4 M held by the MRNF.

22.6 Production Cost and All-in Sustaining Cost

A summary of the operating costs is presented in Table 22.5. Total cash costs are in USD and calculated on per ounce of gold using an exchange rate of 0.77 USD:CAD and the cost of mining, processing, technical services, environment, G&A, refining cost and royalties.

The LOM operating cost per ounce is USD 1,008.82 /oz. The LOM all-in sustaining cost (AISC) per ounce is USD 1,120.43 /oz.

Table 22.5 – Production cost summary

Description	Unit	LOM value
Metal production		
Gold	koz	181.3
Cost and Royalty		
Direct operating cost	$M	218.3
Refining and smelting	$M	0.9
Royalty	$M	8.5
Cash cost and AISC		
Gold price	USD/oz	1,800.00
Exchange rate	USD:CAD	0.77
Cash cost	**USD/oz**	**1,008.82**
AISC	**USD/oz**	**1,120.43**

22.7 Sensitivity Analysis

A sensitivity analysis was carried out on the financial model to assess the variations on the IRR and the NPV, using a 5% discount rate, varying the parameters of gold price, operating costs, capital costs and revenues (variation of the head grade).

Table 22.6 shows the pre-tax and after-tax values of the NPV and IRR for a variation in the gold price.

Table 22.6 – NPV and IRR sensitivity analysis results for gold price variations

	Pre-Tax		After Tax	
(US$/oz)	NPV ($M)	IRR (%)	NPV ($M)	IRR (%)
1 550	30,1	21%	22,6	18%
1 600	39,6	25%	30,4	21%
1 650	49,1	29%	36,6	25%
1 700	58,6	33%	42,5	28%
1 750	68,1	37%	48,4	30%
1 800	77,5	41%	54,4	33%

	Pre-Tax		After Tax	
(US$/oz)	NPV ($M)	IRR (%)	NPV ($M)	IRR (%)
1 850	87,0	45%	60,2	36%
1 900	96,5	48%	66,1	39%
1 950	106,0	52%	72,0	42%
2 000	115,5	55%	77,9	44%
2 050	124,9	59%	83,8	47%

Table 22.7 shows the pre-tax and after-tax values of the NPV and IRR for a variation in the capital cost.

Table 22.7 – NPV and IRR sensitivity analysis results for capital costs variations

	Pre-Tax		After Tax	
(%)	NPV ($M)	IRR (%)	NPV ($M)	IRR (%)
-50%	107,4	72%	75,4	60%
-40%	101,4	64%	71,2	53%
-30%	95,4	57%	67,0	47%
-20%	89,5	51%	62,8	42%
-10%	83,5	46%	58,6	37%
0%	77,5	41%	54,4	33%
10%	71,6	37%	50,1	30%
20%	65,6	33%	45,9	27%
30%	59,6	30%	41,7	24%
40%	53,7	27%	37,5	21%
50%	47,7	24%	33,3	19%

Table 22.8 shows the pre-tax and after-tax values of the NPV and IRR for a variation in the operating cost.

Table 22.8 – NPV and IRR sensitivity analysis results for operating costs variations

	Pre-Tax		After Tax	
(%)	NPV ($M)	IRR (%)	NPV ($M)	IRR (%)
-50%	179,7	88%	118,5	70%
-40%	159,3	78%	105,7	62%
-30%	138,8	69%	92,9	55%
-20%	118,4	59%	80,1	48%
-10%	98,0	50%	67,2	40%
0%	77,5	41%	54,4	33%
10%	57,1	32%	41,4	26%

	Pre-Tax		After Tax	
(%)	NPV ($M)	IRR (%)	NPV ($M)	IRR (%)
20%	36,7	23%	27,8	19%
30%	16,2	13%	10,5	11%
40%	(4,2)	3%	(8,8)	0%
50%	(24,6)	0%	(27,7)	0%

Table 22.9 shows the pre-tax and after-tax values of the NPV and IRR for a variation in the revenues.

Table 22.9 – NPV and IRR sensitivity analysis results for revenues variations

	Pre-Tax		After Tax	
(%)	NPV ($M)	IRR (%)	NPV ($M)	IRR (%)
-50%	(95,1)	0%	(96,5)	0%
-40%	(60,6)	0%	(62,3)	0%
-30%	(26,0)	0%	(28,2)	0%
-20%	8,5	10%	4,3	8%
-10%	43,0	27%	32,7	23%
0%	77,5	41%	54,4	33%
10%	112,1	54%	75,8	43%
20%	146,6	66%	97,2	53%
30%	181,1	78%	118,6	62%
40%	215,6	89%	139,9	70%
50%	250,2	100%	161,2	78%

Figure 22.1 depicts the graphical representation in the pre-tax value of the NPV for a variation in the price of gold, capital costs, operating costs and revenue. It should be noted that the curves for revenue and the price of gold overlap.

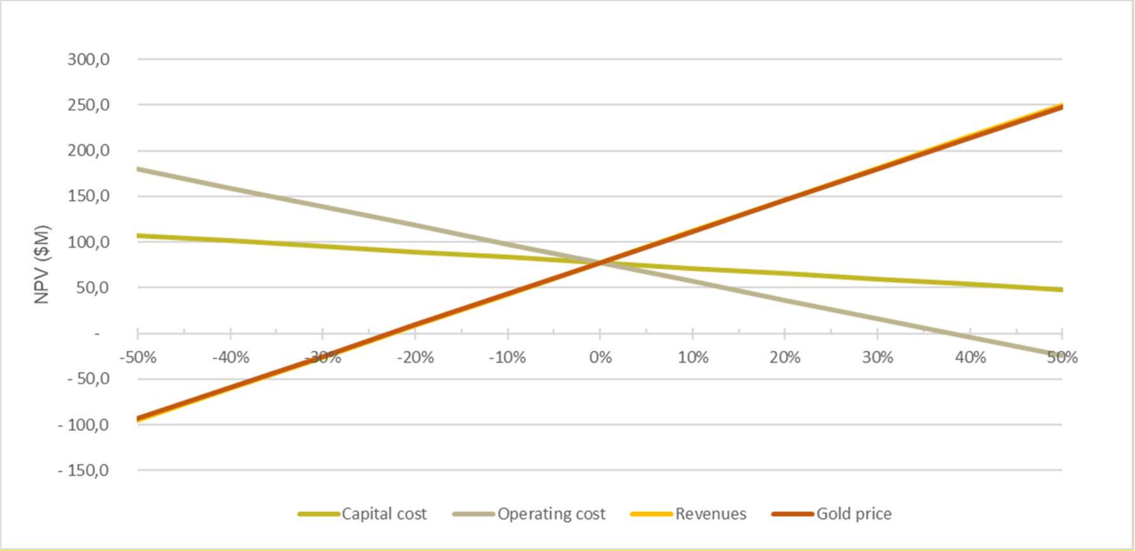


Figure 22.1 – Sensitivity analysis on NPV (pre-tax)

Figure 22.2 depicts the graphical representation in the pre-tax value of the IRR for a variation in the price of gold, capital costs, operating costs and revenue. It should be noted that the curves for revenue and the price of gold overlap.

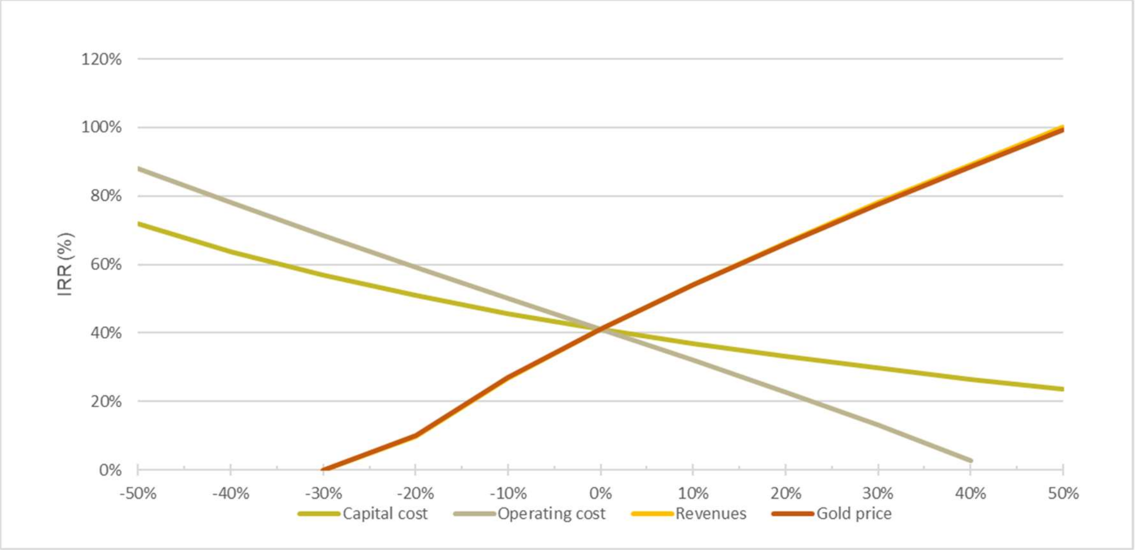


Figure 22.2 – Sensitivity analysis on IRR (pre-tax)

Figure 22.3 depicts the graphical representation in the after-tax value of the NPV for a variation in the price of gold, capital costs, operating costs and revenue. It should be noted that the curves for revenue and the price of gold overlap.

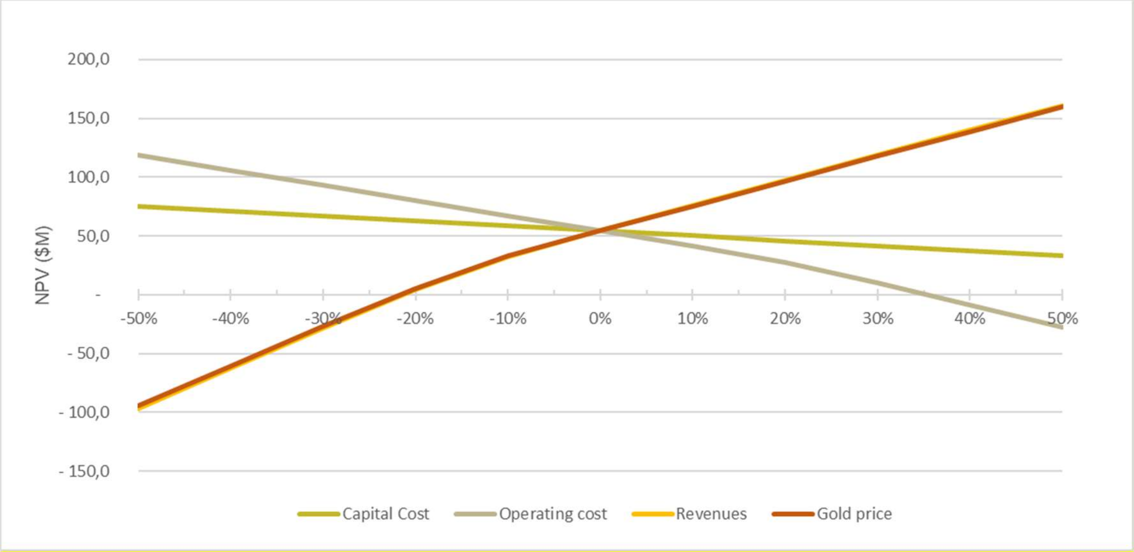


Figure 22.3 – Sensitivity analysis on NPV (after-tax)

Figure 22.4 depicts the graphical representation in the after-tax value of the IRR for a variation in the price of gold, capital costs, operating costs and revenue. It should be noted that the curves for revenue and the price of gold overlap.

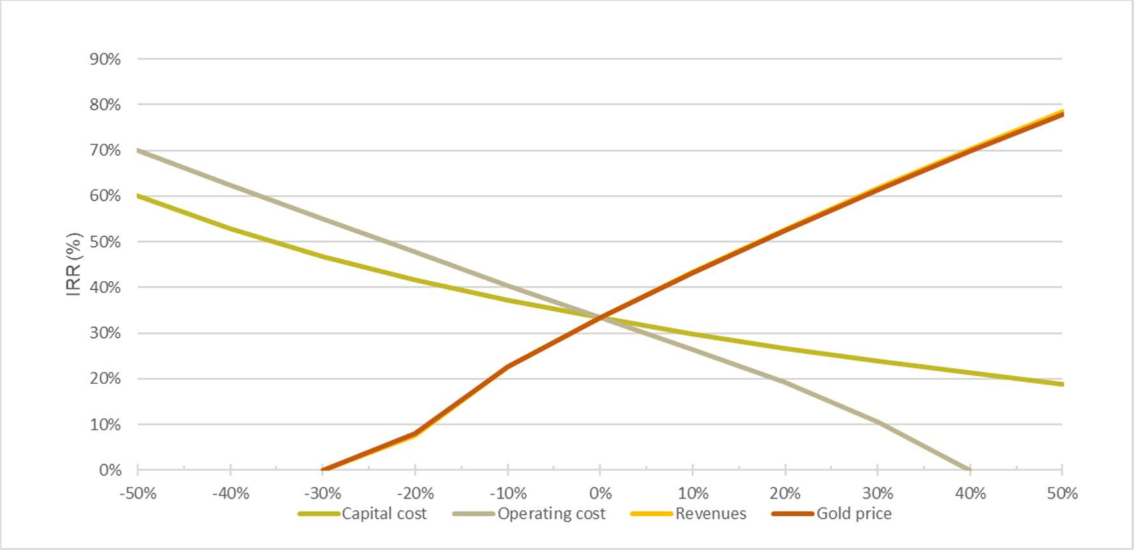


Figure 22.4 – Sensitivity analysis on IRR (after-tax)

The analysis of the results indicates that the value of the NPV as well as the IRR before and after taxes is more sensitive to a change in revenues (variation of the head grade) and gold price (as indicated by both results being), than, to a lesser extent, the impact of a change in operating cost and finally the capital cost with the least impact.

23. ADJACENT PROPERTIES

The Property is surrounded by other exploration claims at early stages of exploration. Except for Muzhu Mining, Globex and Midland Exploration (as described below), most exploration properties neighbouring the Property are held by independent owners (Figure 23.1).

None of the adjacent properties are relevant to the Report or to the progress of the Property.

The information presented herein about mineralization on adjacent properties is not necessarily indicative of mineralization on the Property. None of the Report Authors have verified any mineral resource estimates or published geological information pertaining to the adjacent properties.

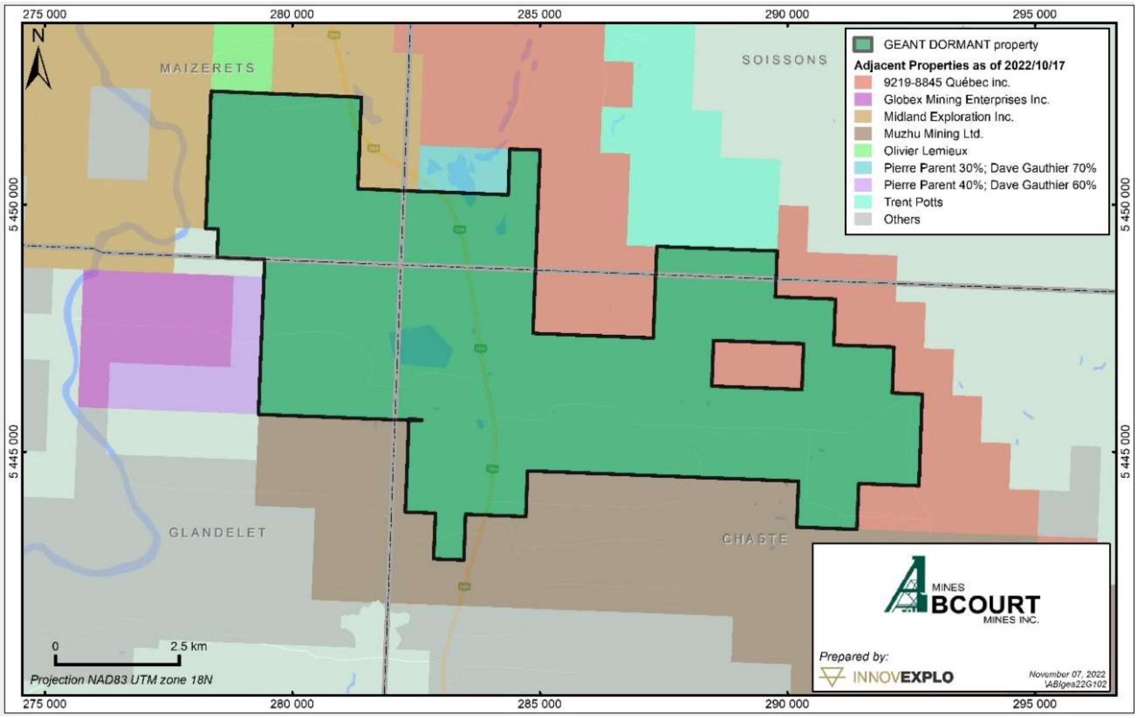


Figure 23.1 – Map showing outlines of properties in the immediate vicinity of the Project

23.1 Muzhu Mining – Sleeping Giant South

The following description is based on information obtained from the website of Muzhu Mining Ltd.

23.1.1 Overview

- 100% ownership of the Sleeping Giant South property
- Contiguous claims (15,000 acres) in the Quevillon gold camp
- NI 43-101 technical report completed in December 2020
- Located 500 m south of the Sleeping Giant mine (Abcourt)

23.1.2 Targets

The Sleeping Giant South property is underlain by a highly prospective geological unit in the Abitibi Greenstone Belt and hosts major deformation corridors. It has been historically underexplored for base- and precious-metals. Historical assay results returned Cu-Zn-Au-Ag values, and historical drilling returned visible sulphide mineralization. Muzhu Mining claims that a nearby VMS showing and Cu-Zn-Au-Ag mineralization support the discovery potential underlying their property.

23.1.2.1 Geochemical Targets

- In the west: elevated copper values (up to 152 ppm Cu) in basalt-andesite assemblages.
- In the centre: elevated copper values (up to 377 ppm Cu) in the same basalt-andesite assemblage.
- In the east: elevated Au-Cu-Zn values (up to 10 ppb Au, 131 ppm Cu and 201 ppm Zn) in gabbro, felsic tuff and basalt-andesite assemblages.

23.1.2.2 Geophysical and Drilled Anomalies

Muzhu has flown a low-altitude, high-resolution magnetic response survey over their entire property using tight line spacing. This configuration has seen much success with neighbouring companies in the district. The total survey length was 1,620 line-km.

23.2 Midland Exploration – Nickel Square

The following description is based on information obtained from the Midland Exploration Inc. (“Midland”) website.

Midland owns a 100% interest in the Nickel Square property (460 claims, 260 km^2) located between Amos and Matagami.

The Nickel Square property is of interest to Midland for its underexplored potential for Ni-Cu-Co-PGE mineralization. The property is underlain by the Maizerest Intrusions, a series of ultramafic intrusions locally associated with untested historical electromagnetic conductor anomalies.

23.3 Globex - Napping Dwarf Property

According to the website of Globex Mining Enterprises Inc., the Napping Dwarf gold property is located in Glandelet Township (NTS 32E/01). Aside from a vertical gradient magnetic map showing the property's location, no other information is provided on the company's website.

24. OTHER RELEVANT DATA AND INFORMATION

The Report Authors are not aware of any other relevant data and information that could have a significant impact on the interpretation and conclusions presented in the Report.

There is no additional information or explanation necessary to make the Report understandable and not misleading.

25. INTERPRETATION AND CONCLUSIONS

The QP have provided the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for the Report.

25.1 Property Description and Ownership

The Property is owned 100% by Abcourt and is located 80 km north of the town of Amos northwestern Quebec. It is accessed from provincial highway Route 109, which runs N-S through the eastern part of the Property and connects Amos to Matagami. A 1-km gravel road leads westward from Route 109 to the mine-site gatehouse. A network of forestry roads provide access to other parts of the Property (Figure 4.1).

25.2 Geology

The 2023 PEA presents the first 3D interpretation of the Sleeping Giant deposit and the first pre-economic evaluation since the most recent 3D interpretation of the Sleeping Giant deposit.

The main objective of the current mandate was to assess the potential economic viability of the 2022 MRE.

The 2022 MRE was prepared using 3D block modelling and the ID2 interpolation method. Genesis was used to create the 3D mineralized veins, Geovia Surpac 2022 was used to perform the interpolation, and Deswik.SO was used to optimize the mineable stopes above the determined cut-off grade.

The final resources of the 2022 MRE are similar to those of the 2019 MRE; however, 3D shapes of mineable stopes have been implemented in the 2022 MRE, yielding a much more robust, albeit more restrictive, MRE.

The QP reviewed all information contained in the database provided by the Issuer.

25.3 Mineral Resource Estimate

The 2022 MRE includes all blocks (“must-take blocks”) falling within a potentially mineable shape to satisfy the criterion of reasonable prospects for eventual economic extraction, as specified by CIM Guidelines (2019). It should be noted that these guidelines were introduced after the 2019 MRE.

Table 25.1 – Mineral Resource Estimate for the Sleeping Giant Mine (effective as of December 12, 2022)

	Potential Long Hole (Cut off at 4.25 g/t Au)			Potential Room and Pillar (Cut off at 5.00 g/t Au)			Total		
	Tonnes	Grade (Au g/t)	Ounces AU (oz)	Tonnes	Grade (Au g/t)	Ounces AU (oz)	Tonnes	Grade (Au g/t)	Ounces AU (oz)
Indicated Resources	677,000	7.03	153,000	78,000	7.98	20,000	755,000	7.14	173,300
Inferred Resources	677,000	8.13	177,000	207,000	10.67	71,000	884,000	8.74	248,300

Notes to the 2022 MRE:

1. The independent and qualified persons for the 2022 MRE, as defined by NI 43-101, are Olivier Vadnais-Leblanc (P.Geo.) and Eric Lecomte (P.Eng.), both from InnovExplo Inc.
2. These mineral resources are not mineral reserves because they do not have demonstrated economic viability. The results are presented undiluted and are considered to have reasonable prospects of economic viability. The 2022 MRE follows CIM Definition Standards (2014) and CIM Guidelines (2019).
3. The estimate encompasses 846 mineralized zones modelled using a minimum geological width of 0.5 m in Genesis software.
4. A density value of 2.85 g/cm^3 (based on measurements and mine and mill reconciliation) was assigned to all mineralized zones.
5. High-grade capping supported by statistical analysis was established at 95 g/t Au for all mineralized zones and applied to the composite data. Composites (0.5 m) were calculated within the zones using the grade of the adjacent material when assayed or a value of zero when not assayed.
6. The RPEEE requirement (Reasonable Prospect of Eventual Economical Extraction) is fulfilled using cut-off grades based on reasonable mining parameters, locally constrained within Deswik Stope Optimizer shapes using a minimum mining width of 1.7 m for both potential methods. It is reported at a rounded cut-off grade of 4.25 g/t Au using the long-hole ("LH") method and 5.0 g/t Au using the room and pillar ("R&P") method. The cut-off grades were calculated using the following parameters: mining cost = C$213.96/t (LH) to C$261.56/t (R&P); processing cost = C$35.10/t; G&A = C$22.09/t; gold price = US$1,650.00/oz and USD:CAD exchange rate = 1.30. The cut-off grades should be re-evaluated in light of future prevailing market conditions (metal prices, exchange rates, mining costs etc.).
7. The estimate was completed using a sub-block model in Surpac 2022. A 4m x 4m x 4m parent block size was used (1m x 1m x 1m sub-blocked). Grade interpolation was obtained by Inverse Distance Squared (ID2 using hard boundaries.
8. The mineral resource estimate is classified as Indicated and Inferred. The Inferred category is defined with a minimum of three (3) drill holes within the areas where the drill spacing is less than 75 m and shows reasonable geological and grade continuity. The Indicated mineral resource category is defined with a minimum of four (4) drill holes within the areas where the drill spacing is less than 30 m and shows reasonable geological and grade continuity.
9. The number of metric tons was rounded to the nearest hundred, following the recommendations in NI 43-101, and any discrepancies in the totals are due to rounding effects. The metal contents are presented in troy ounces (tonnes x grade / 31.10348) rounded to the nearest hundred.
10. The independent and qualified persons for the 2022 MRE are not aware of any known environmental, permitting, legal, political, title-related, taxation, socio-political, or marketing issues that could materially affect the estimate.

25.4 Infrastructure

In August 2022, Abcourt commissioned a firm specializing in cleaning gold mills. Cleaning of the Sleeping Giant processing plant began in September 2022 and was still underway as at the effective date of the Report. Abcourt planned to continue the cleaning program until February 2023, when the plant will enter maintenance and preparation mode until mining operations can feed the plant at a constant rate.

The Sleeping Giant mill has a capacity of 750 tpd. Mineralized material can be transported directly to the ore pass feeding the crusher, or stored on the stockpile, which has a capacity of 10,000 t. The plant is used for grinding, leaching and desorption, gold electrolysis and refining. The mill's waste is sent as pulp to the TSF through an 8-inch-diameter pipe.

The TSF consists of three ponds. The first is used to decant the pulp, and the second recovers water for reuse as process water in the plant. The surplus is sent to the polishing pond for processing (once a year). Water from this polishing pond is sent to two other ponds, where it is analyzed before being discharged into the environment after ensuring it complies with the rules and by-laws of the Ministry of the Environment.

Hydro-Québec supplies the mine with electricity. The capacity is greater than 5,000 KVA, which is adequate for surface and underground operations.

The ventilation shaft and surface shaft shelters are operational. The service building, mechanical workshop, electrical workshop, equipment warehouse, hoist room, compressor room, generator, water pump for fire, drinking water pumping system and dry room are all currently functional.

The headframe and service shaft (Shaft #2) provides access to the underground infrastructure. The service shaft accesses 22 levels. The first level is 50 m from the surface, and the last is at a depth of 1,175 m. The ventilation shaft (Shaft #1) is used to bring fresh air into the mine. It can also be used as an emergency exit for employees.

The underground mine pumping system is functional. Underground water is pumped into basins designed for that purpose. The waste pile receives low-grade development material. Non-mineralized rocks on the waste rock pile are mostly used as construction material for dikes in the TSF, as the rock is not acid-generating.

25.4.1 Site infrastructures and facilities

Site infrastructures are basically in good conditions. The ore silo and loading stations need to be reconditioned along with the major equipment in the ore treatment plant. Electrical distribution must be modified to comply with code, mainly with respect to ground fault protection.

A new camp complex should provide comfort and rest for the workforce while on site. It may also help attract labour, which is in short supply in the area due to the abundance of active mining operations. The design of a new wastewater treatment system is required to accommodate the increased volume of wastewater to treat.

25.4.2 Tailing storage facilities

The current TSF is located north of the processing plant and comprises three (3) cells: Cell 1 (deposition pond), Cell 2 (recirculation pond) and Cell 3 (polishing pond).

Upgrades to the TSF will be required in order to be able to store the mine tailings generated by the 720,000 t of mined material projected to be milled in the processing plant. Some maintenance work and completion of certain structures in Cell 1 are required. It is also required to split Cell 2 into two parts. This modification would allow the storage of the mine tailings in the future Cell 2A and the remaining Cell 2 would continue to be used for recirculation of mining process water to the processing plant. The lifts of the various dikes of Cell 2A will be built using tailings and waste rock. Although the TSF is compliant with regulations for the design flood water storage capacity, the current water management strategy brings certain operational risks. These risks can be mitigated by creating additional water storage capacity in Cell 2 by dredging around 120,000 m^3 of tailings.

The treatment of the mining wastewaters from the TSF before their discharge into the environment will be carried out in a manner similar to that of the most recent years of operation of the site. For the resumption of operation of the Sleeping Giant mine, maintenance work or replacement is planned on pumps and nozzles currently being used.

25.5 Mineral Processing and Metallurgical Testing

The Sleeping Giant mine began his operation in 1988 and was last operational in 2014. From August 2016 until 2022, the processing plant was fed by ore from nearby mines. A recovery model was developed based on historical head-grade data collected during past processing of Sleeping Giant ore.

Historical daily processing reports show that the processing plant was able to consistently process between 700 and 750 t/d of ore.

25.6 Recovery Methods

Mined material is processed using two stages of crushing, rod mill grinding, ball mill grinding and a Carbon-In-Leach gold plant.

The gold recovery is based on a model that was developed from mill performances when processing Sleeping Giant ore. This model predicts the gold recovery based on head grade. With a head grade of 8.1 g/t, the expected recovery is 96.7%.

The capacity of the processing plant is between 700 and 750 t/d, which is higher than the projected expected processing tonnage of 350 t/d on a 12 hours daily shift for the Project.

There is no modification in the plant throughput, equipment size, number of equipment or ore that are expected to change the recovery model or the plant capacity.

25.7 Mining Methods

The proposed mining at Sleeping Giant will employ a combination of long hole, room & pillars, and shrinkage mining methods. Shaft rehabilitation is planned to begin in 2024, with rehabilitation of existing levels continuing in the succeeding months. Underground mined material will be processed at a rate of approximately 350 tpd.

The underground operation is a conventional mining operation, using track drift to haul mineralized material to the existing ore pass system. To reach the optimized stopes, new drifts are excavated from existing levels. Additionally, some new levels will be developed near the bottom of the shaft (below N-1060). The current operation has a remaining mine life through to Q4 2030.

The Project will involve an underground operation with a proposed mining method optimized to the deposit geometry (narrow veins) and previously used production techniques. Mining voids will not be backfilled, continuing the previous approach.

A total of 720,000 tonnes of mineralized material is planned to be mucked out of the mine by August 2030, on a seven (7) year production plan. Approximately 43 % of the mineralized material will come from shrinkage stopes. Room-and-pillar stopes and long-hole stopes will account respectively for 30 % and 27 % of the production plan.

Abcourt plans to continue operating providing that additional resources are defined by an on-going exploration program, or through an increase in the price of gold.

The development mine plan is presented in Table 25.2 and the production mine plan is Table 25.3.

Table 25.2 – Mine plan - Development

Development	Unit	2024	2025	2026	2027	2028	2029	2030	Total
Rehabilitation	m	7,460	930	2,060	780	2,600	2,260	0	16,090
Horizontal development	m	3,370	3,290	2,560	2,100	1,930	1,950	250	15,440
Vertical development	m	570	620	530	490	540	440	180	3,360
Waste Produced	t	67,700	54,910	55,010	34,490	37,670	37,270	4,790	291,860

Table 25.3 – Mine plan - Production

Production	Unit	2024	2025	2026	2027	2028	2029	2030	Total
Tonnes Mined	kt	20.5	110.7	123.6	127.5	126.2	127.5	84.2	**720**
Gold Grade	g/t	7.25	6.76	8.37	9.63	7.31	7.68	9.20	**8.10**
Gold (mined)	koz	4.8	24.0	33.2	39.5	29.7	31.5	24.9	**188**
Stockpile Inventory	kt	81.8	29.3	-	-	-	-	-	**111**
Gold Grade	g/t	7.12	7.25	-	-	-	-	-	**7.15**
Gold (mined)	koz	18.7	6.8	-	-	-	-	-	**26**
Tonnes Milled	kt	3.5	127.7	123.6	127.5	126.2	127.5	84.2	**720**
Mineralized material per day (average)	t	10	360	348	359	356	359	237	**336**

25.8 Environmental, Social, Community, and Reclamation / Closure

In June 2016, the cession of Certificates of Authorization (CofA) from previous owners to Abourt was completed.

Sleeping Giant mine is subjected to Canadian and Québec requirements in term of monitoring studies and environmental controls. The mining site is subjected to the federal metal and diamond mining effluent regulations.

As the proposed Project remains under the thresholds mentioned in the EQA for an environment impact assessment, it does not trigger the environmental impact assessment procedure. The Project is also under the thresholds of the federal legislation (*Impact Assessment Act and Physical Activities Regulations*) and as such, does not trigger a federal impact assessment study.

The proposed Project falls under the EQA and the *Regulation respecting the regulatory scheme applying to activities on the basis of their environmental impact* (Q-2, r. 17.1). The permitting process is triggered by the EQA Section 22. Abcourt currently has most of the CofA allowing it to mine and process mineralized material; however, a request for modification of certain existing authorizations will have to be presented concerning work related to drinking water, wastewater treatment and the tailing facility.

The mine-site is also subject to a depollution attestation under Division III of the EQA. Technical advice received from the MELCCFP in March 2020 related to the depollution attestation mentioned that new monitoring wells to assess the groundwater quality is required, considering that some values are exceeding applicable criteria in monitoring wells not installed properly.

Complementary work is required related to wastewater treatment system and groundwater wells, in addition to a hydrogeological study in connection with the depollution attestation.

The study area is located in the southern part of the territory of the Eeyou Istchee James Bay Regional Government on Category III Lands. The Regional Government is governed under the laws of Québec and exercises the jurisdictions, functions, and powers over Category III Lands in the Eeyou Istchee James Bay Territory. So far, there has been no negotiation nor any agreement involving Abcourt and any group, organization, or First Nations regarding the Project.

In 2018, an updated version of the rehabilitation and restoration plan for the Sleeping Giant mine was filed to the MRNF by Abcourt. Subsequently, Abcourt received comments from the MRNF on the closure plan. Their responses were filed with the MRNF in February 2023. A revised cost estimate for site closure was included. Abcourt is currently waiting the feedback from the MRNF. The restoration concept approved by the MRNF in the previous versions of the closure plan is currently maintained.

25.9 Financial Analysis

The economic/financial evaluation of the Project was carried out using the discounted cash flow approach on a pre-tax and after-tax basis. The gold price was derived from a consensus based on long-term average commodity prices in USD and expenditure estimates in the currency in which they were made. An equivalent exchange rate of 0.77 USD to 1.00 CAD was used to convert the market price of raw materials into CAD. No provision has been made for the effect of inflation. Current Canadian corporate tax regulations have been considered for the assessment of federal mining tax liabilities and the most recent provincial tax regulations have been used to access the Québec mining tax liabilities.

For the pre-tax base case, the financial model resulted in an IRR of 41.1% and a NPV of $77.5 M with a discount rate of 5%. The payback period following the start of the commercial production is 2.1 years. On an after-tax basis, the financial model resulted in an IRR of 33.2% and a NPV of $33.2 M with a discount rate of 5%. The payback period following the start of the commercial production is 2.2 years.

25.10 Risks and Opportunities

Table 25.4 identifies the significant internal risks, potential impacts and possible risk mitigation measures that could affect the future economic outcome of the Project. 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 25.5. Further information and study are required before these opportunities can be included in the project economics.

Table 25.4 – Risks and Possible Mitigation for the Project

Area	Risks Description and Potential Impact	Possible Mitigation
Geology and Mineral Resources	Locally inaccurate historical assay results, Local bias in grade estimates.	Additional sampling (definition drilling and chip sampling).
	Locally poor geological continuity of veins (i.e., local dislocations, pinch, and swell); Poor geological continuity could negatively affect the accurate localization of mineralized blocks and/or grade estimation.	Acquire additional geological information by drilling (with a locally denser drill pattern) and eventually by direct mapping (when mining exposures become available).
Mining and Mineral Reserves	Local uncertainties about historical mine openings; Local bias in estimates.	Verify local accuracy and refine historical mine openings (where possible).
	Old underground infrastructure: Rehabilitation work is needed to restart operations. Inadequacies have been identified in the existing infrastructure.	Identify places where access to resources is limited. Verify the status of historical stopes (open, backfill).
Process and Metallurgy	Lower Gold recovery than predicted that will negatively impact the project profitability	Follow-up of the production and the gold recovery to meet the expected gold recovery. Increase leaching time with air injection during plant shutdown as the plant will operate only 12 hours per day
	Tailings management facility ("TMF") reaching design capacity; Investment may be required to increase TMF capacity, which could lead to delays in mine production	Address the specific TMF studies needed and conduct the necessary work
Infrastructures	Power outage on site	U/G generator (primarily for dewatering)
	Risk associated with a main transformer failure	Ensure having a spare transformer on site
	Risks associated with water management at the TSF	Develop operational procedures for water management and risk management procedures
	Delay to construct the Cell 2A due to regional labour constraints	Develop constructability strategy during the next study phase
Environmental risks	Social acceptability related to new or modified infrastructures (i.e. camp site, wastewater treatment facility) is an inherent risk for all mining projects. It could affect the Project's permitting and development schedule.	Maintain and implement a pro-active and transparent strategy to identify all stakeholders and establish a communication plan.
	Delay in obtaining some permits or CofA (ministerial authorizations), which may impact the	Start the various applications as soon

Area	Risks Description and Potential Impact	Possible Mitigation
	project schedule	as possible
	Technical advice received from the MELCCFP in March 2020 related to the depollution attestation requiring new monitoring wells to assess the groundwater quality	Perform a hydrogeological study in connection with the depollution attestation
	Updated 2018 closure plan not approved by the MRNF. A revised cost estimate for site closure submitted in Feb 2023	Execute complementary studies included in the closure costs to confirm the actual closure concept, if required
Others	Inability to attract experienced professionals; The ability to attract and retain competent, experienced professionals is a key factor to success.	An early search for professionals will help identify and attract critical people through all project phases, from early exploration to more advanced.

Table 25.5 – Opportunities for the Project

Opportunities	Explanation	Potential Benefits
Further 3D modelling and interpretation using current drill hole data	By continuing the 3D modelling, more veins could be estimated without more drilling.	Increase resources
Drill areas with strong possibilities of expanding known mineralized zones	Many areas in the deposits are not drilled on a tight grid. Mineralized veins could easily be expanded with short drill holes.	Increase resources
Understand the structural pattern in certain areas	Some areas of the deposit contain gold, but drill holes in those areas are not properly oriented to ensure a proper understanding of the structural pattern.	Understand the structure of the mineralization in new areas of the deposits. It could lead to the discovery of new minable zones.
Reduce the thickness of the surface pillar	Due to the lack of data on the structural stability of the rock in the surface pillar, InnovExplo had to exclude the first 50 m from the resources. A geotechnical study could reduce the thickness of the pillar.	Shallow resources in the surface pillar could easily be recovered because the infrastructure is readily available.
Include channel samples in the interpolation procedure to potentially improve the block classification in certain areas	Underground channel samples should be taken in areas where a geologist has observed mineralized zones. This added level of confidence may make it possible to classify blocks as measured in these locations.	Generate resources classified as Measured
Refine the 3D solids used for mining depletion	For the 2022 MRE, InnovExplo created buffers around every underground opening to ensure that previously mined resources are not included in the model. However, those buffers are wide and might exclude resources still available.	By being certain of where the resources are available, InnovExplo could reduce the size of the exclusion buffers and increase resources.

Opportunities	Explanation	Potential Benefits
Optimize logging procedures	Standardize logging procedure by implementing custom tables; Standardize and simplify the list of lithologies	Improve correlation between drill hole and simplify 3D modeling
Relogging and resampling of drill holes	From the new 3D model, evaluate areas where drill core could have been sampled but isn't	Improve the continuity of the mineralized zones
Recovery of economic material at the vicinity of old workings	A safety pillar, including economic material, has been systematically left in place around the old workings.	Potential recovery of this material following geotechnical validation.
Processing ore from other properties to increase feed tonnage to reach plant capacity of 750 t/d	As the processing plant will operate only 12 hours per day to process 350 t/d, ore from adjacent properties could be process at the plant	Increase project profitability
Optimization of the use of tailings for the construction of Cell 2A (TSF)	Construction of cell 2A will benefit from the use of less waste rock	Additional tailing capacity within the TSF. Waste rock could be used elsewhere
Water and tailings management	Expand the TSF with the construction of a new cell (Cell 4) instead of dredging associated with the construction of lift 5 of the Cell 2A for extra retention volume.	Expand the TSF capacity for feed tonnage increase related to other properties and lower risks associated with the design flood for the TSF

26. RECOMMENDATIONS

Based on the results of the 2022 MRE, InnovExplo recommends continuing the assessment of the Sleeping Giant deposit. The proposed work program includes procedural improvements, two phases of drilling and studies that would be required before reopening the mine, as follows:

- Implement a comprehensive QA/QC program for future drill-core sampling;
- Implement a 3D localization system for underground channels;
- Prepare a feasability study for the Sleeping Giant mine;
- Plan a 15,000 m drilling campaign to improve the deposit's inferred resources after identifying targets within the 3D-interpreted geological model.;
- Plan an 8,000 m drilling campaign to convert the deposit's Inferred resources to the Indicated category;
- Plan the reopening of the mine.

26.1 QA/QC program

Set a proper QA/QC validation system for drill-core assays. Use the standard industry methodology to regularly insert blanks, duplicates, and CRM standards during the drill core analytical procedure.

26.2 Underground Channel Localization

The locations of underground channel samples in the Sleeping Giant mine have only been represented on 2D mine plans. A proper 3D location for every new channel should be determined to help future 3D modelling and block classification.

26.3 Exploration Drilling

An infill and exploration drilling program should be performed based on the 2022 geological reinterpretation zones.

- Try different drilling angles to define areas where the mineralization trend is uncertain (Figure 26.1 and Figure 26.2);
- Try to extend known mineralized veins (Figure 26.3 and Figure 26.4).

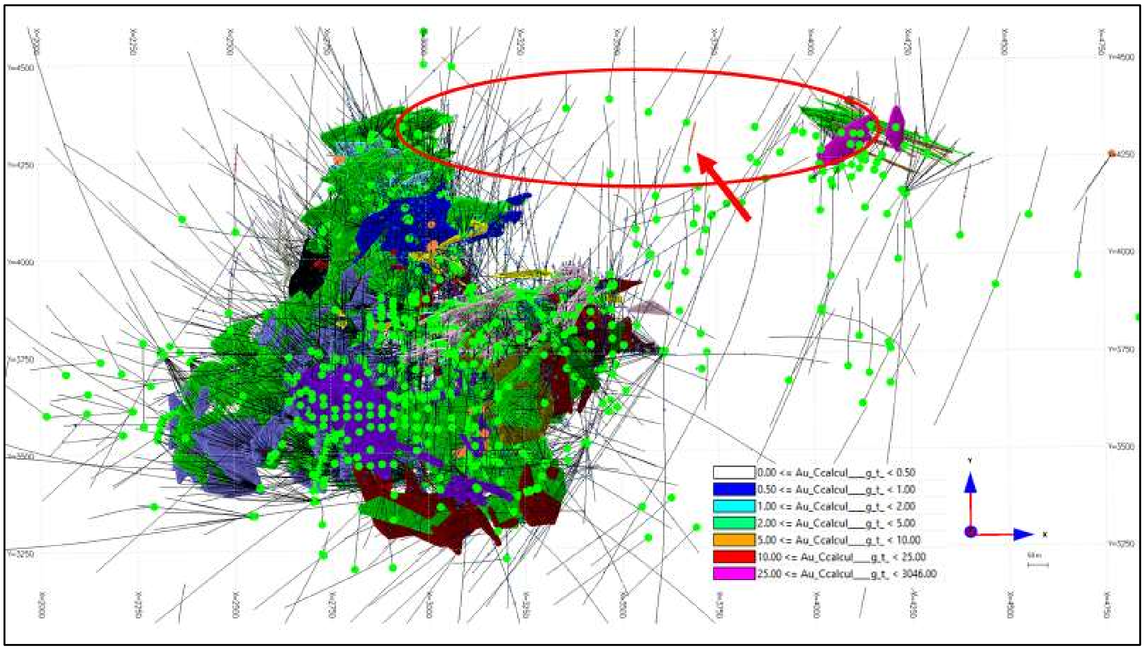


Figure 26.1 – Shallow exploration target (plan view)

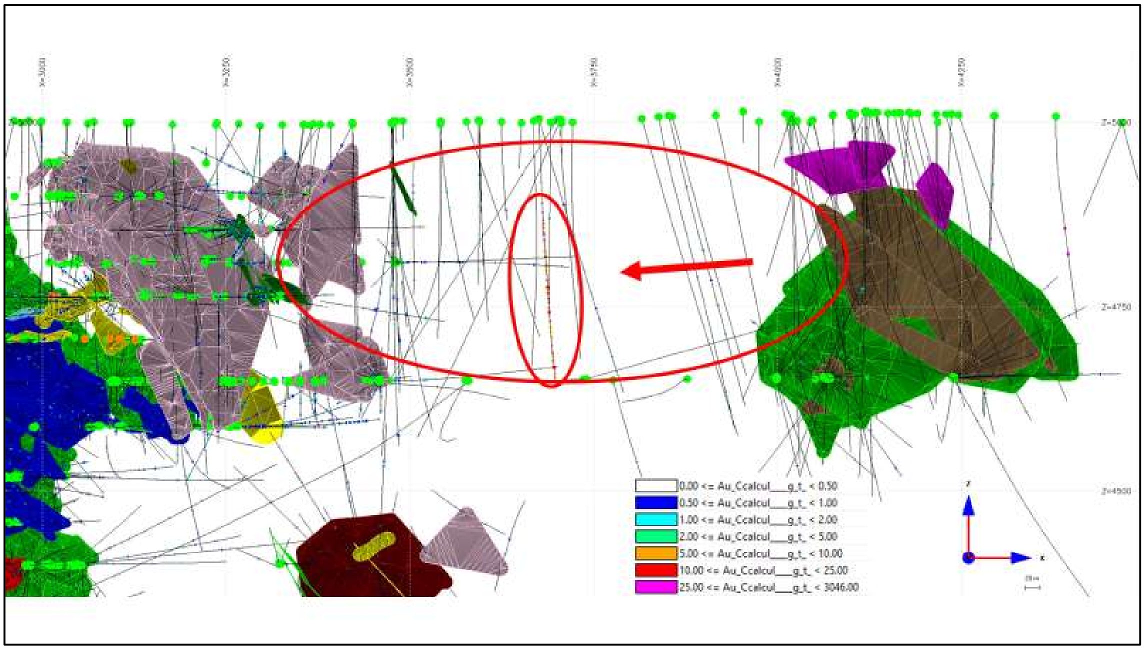


Figure 26.2 – Shallow exploration target (section view)

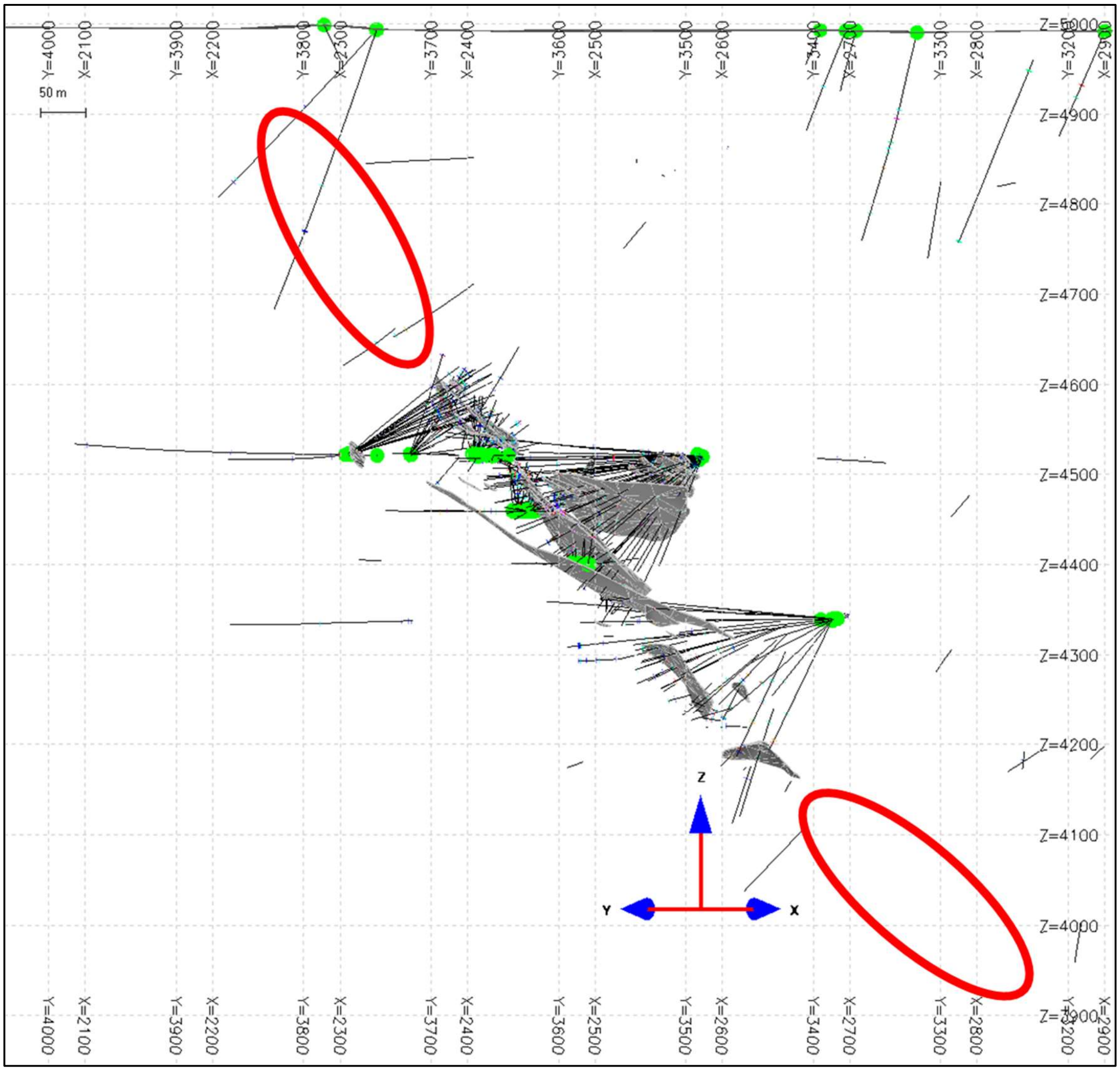


Figure 26.3 – Drilling areas with potential (section view)

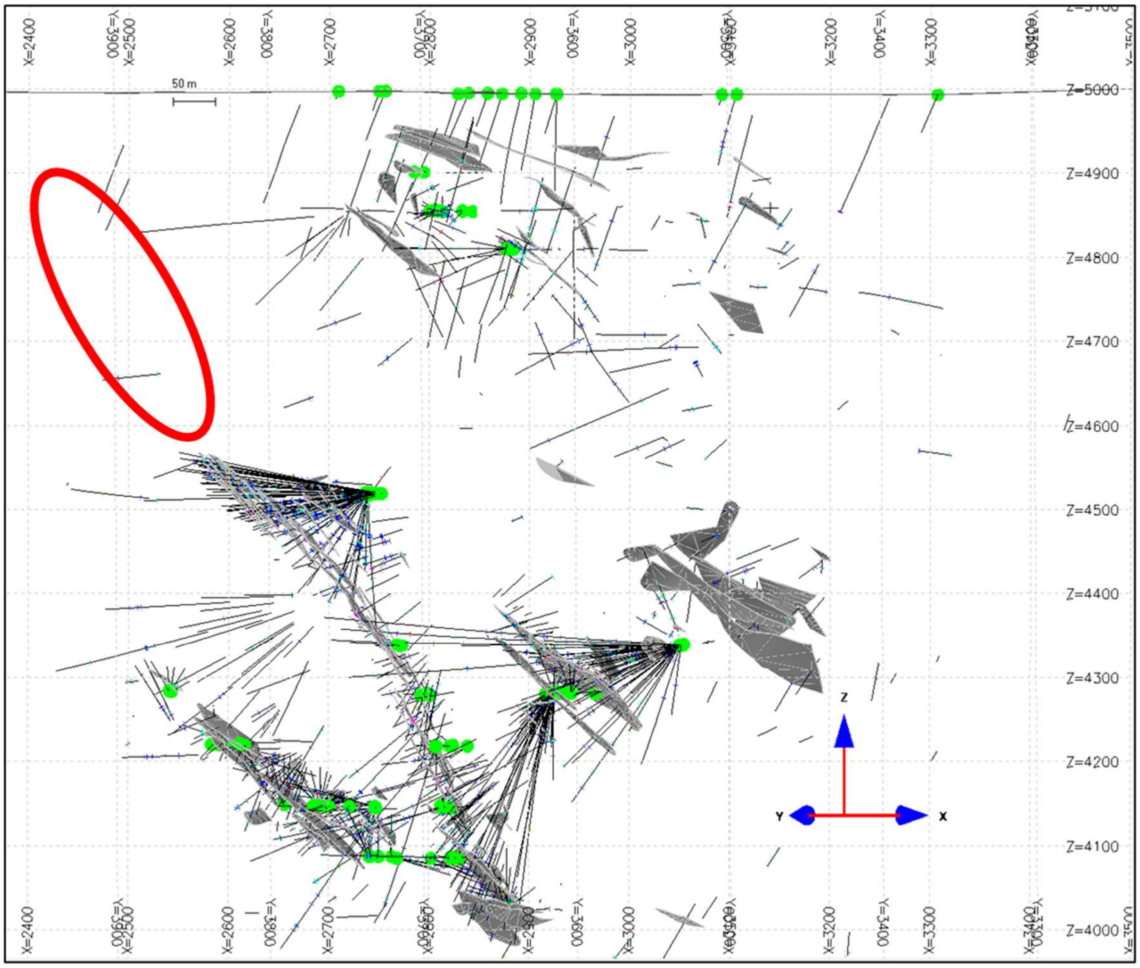


Figure 26.4 – Drilling area with potential (section view)

26.4 Mining Method

To support and optimize the mine planning, it is recommended to:

- Complete additional research and detailed mapping of the old working drift and stopes to gain better understanding of the rehabilitation required and improve mine planning.
- Optimize the mine plan to minimized simultaneous active levels, in order to maximized team efficiencies and lower costs.
- Complete additional engineering work to detail the CAPEX and OPEX of the underground mine; detail the costs linked to rehabilitation of existing drifts.
- Carry out discussions with various suppliers to negotiate agreements.

26.5 Project Infrastructure

- Start planning electrical distribution modifications as soon as possible, for both surface and underground systems;
- Take steps with Hydro Québec to begin the process of modifying the mine-site power line, in order to connect the various camp modules.
- Continue design of a new wastewater treatment system and test the capacity of the groundwater wells;
- Perform a trade-off study for the TSF between lift 5 of Cell 2A and a new cell (Cell 4);
- Feed the next project study with updated construction strategy and contractor prices related to the construction of Cell 2A of the TSF;
- Feed the next project study with validated operational procedures for water management in the TSF.

26.6 Costs Estimate for Recommended Work

InnovExplo and collaborators have prepared a cost estimate for the recommended work program to serve as a guideline. The budget for the proposed program is presented in Figure 26.1. Expenditures are estimated at 4.6 M$ (incl. 15% for contingencies).

Table 26.1 – Estimated Costs for the Recommended Work Program

Work program	Budget cost
Exploration and definition drilling (approx. 23,000 m at $150/m)	$3,450,000
Implement a QA/QC data validation system	$30,000
Channel sampling 3D localization	$20,000
Electrical distribution modifications (Addition of ground fault protection)	$447,000
H-Q power line modification for camp module	$160,000
Optimize mine plan	$50,000
Continue / start permitting initiatives related to the Cell 2A of the TSF to advance toward production	$10,000
TSF: Trade-off study between lift 5 in Cell 2A and a new cell (Cell 4)	$30,000
TSF: Start feasibility study related to the Cell 2A	$50,000
TSF: Engineering/operational procedures	$50,000
Wetland delineation related to the camp infrastructure	$15,000
Groundwater wells: hydrogeological study, tendering and permitting	$150,000
Wastewater treatment system: design, permitting, and specifications	$45,000
Hydrogeological study in connection with the depollution attestation	$140,000
TOTAL (CAD)	$4,647,000

27. REFERENCES

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APPENDIX I – SECTION 18: ELECTRICAL DIAGRAM

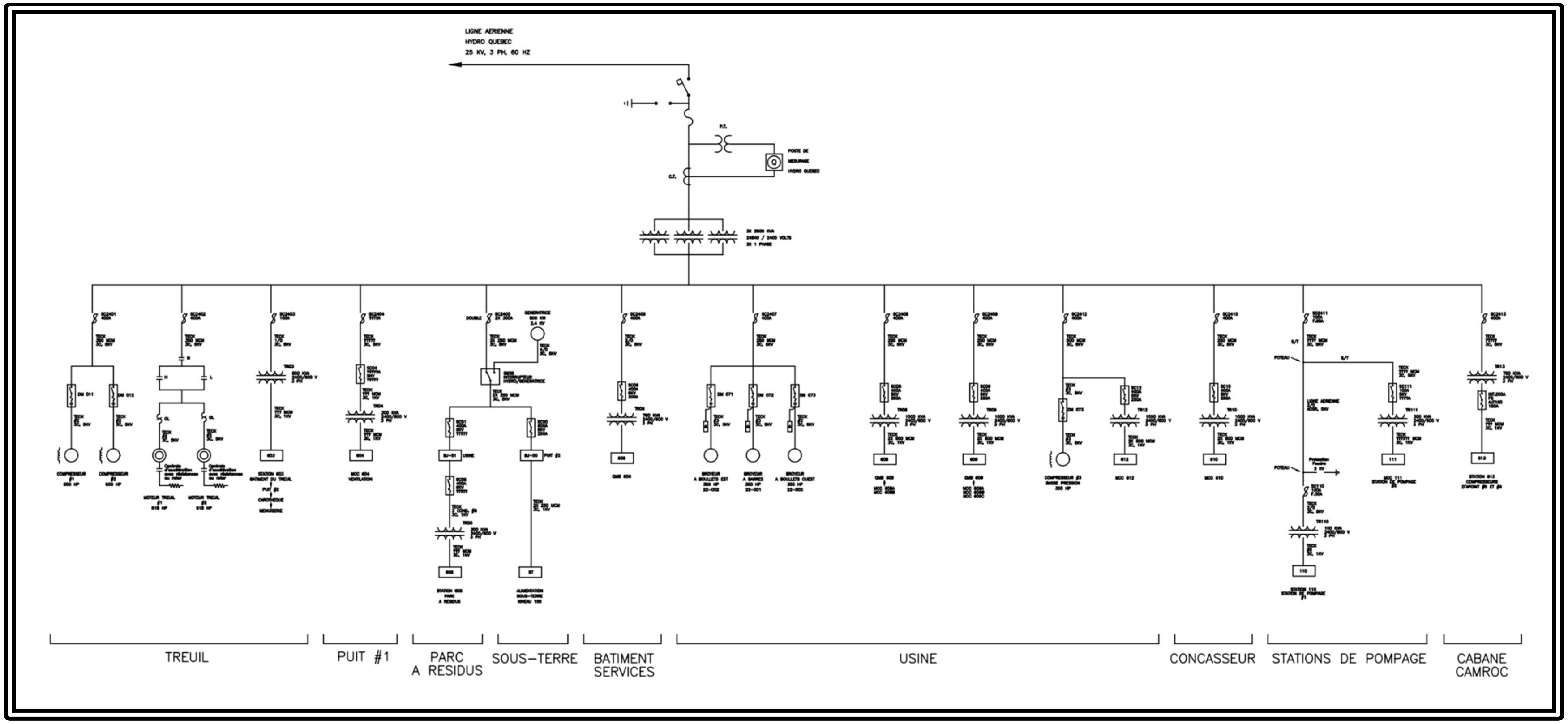

Figure 27.1 – SLD - Surface

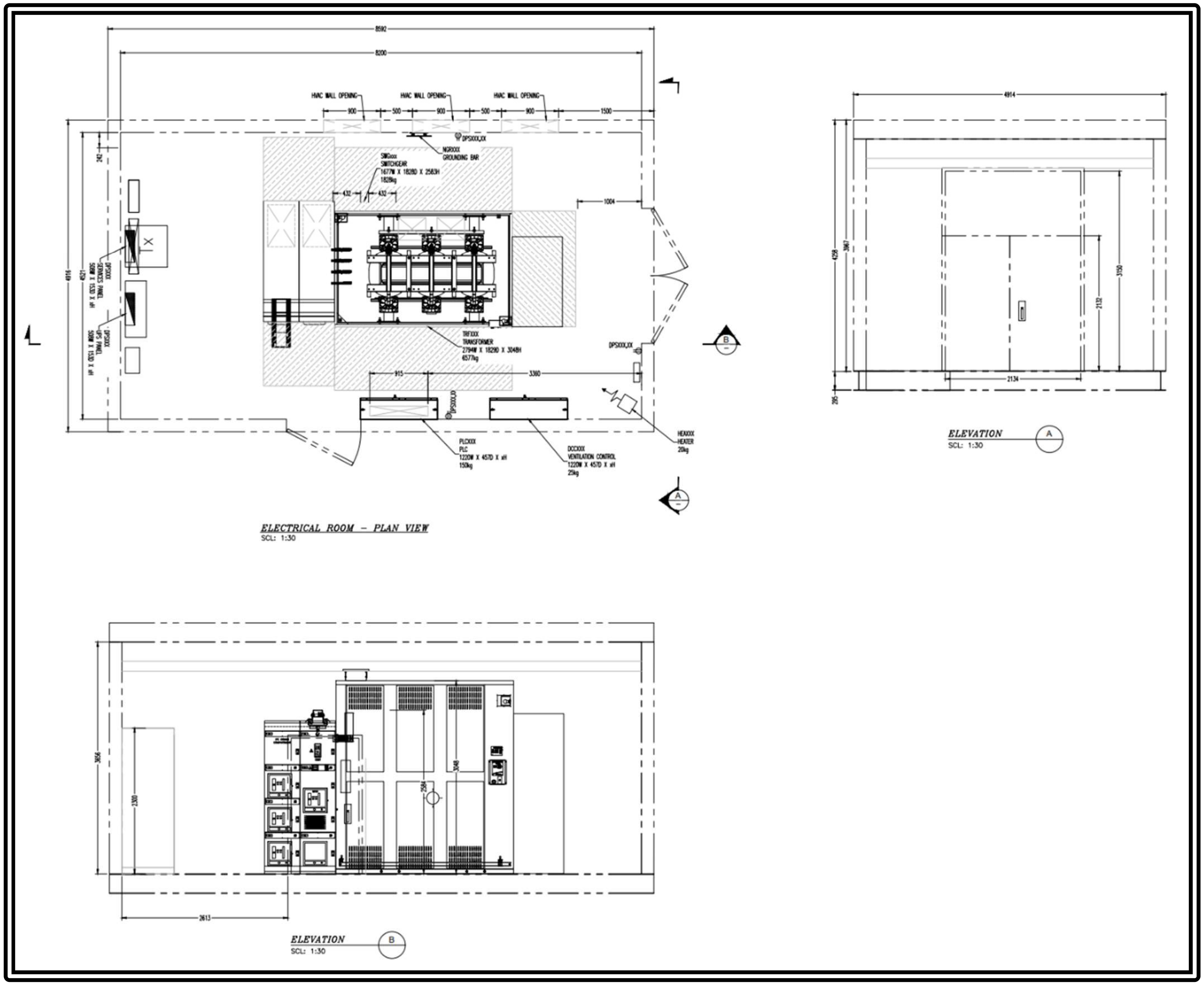

Figure 27.2 – Option 1 GA – Drawing # 0000-E001-0102-0A

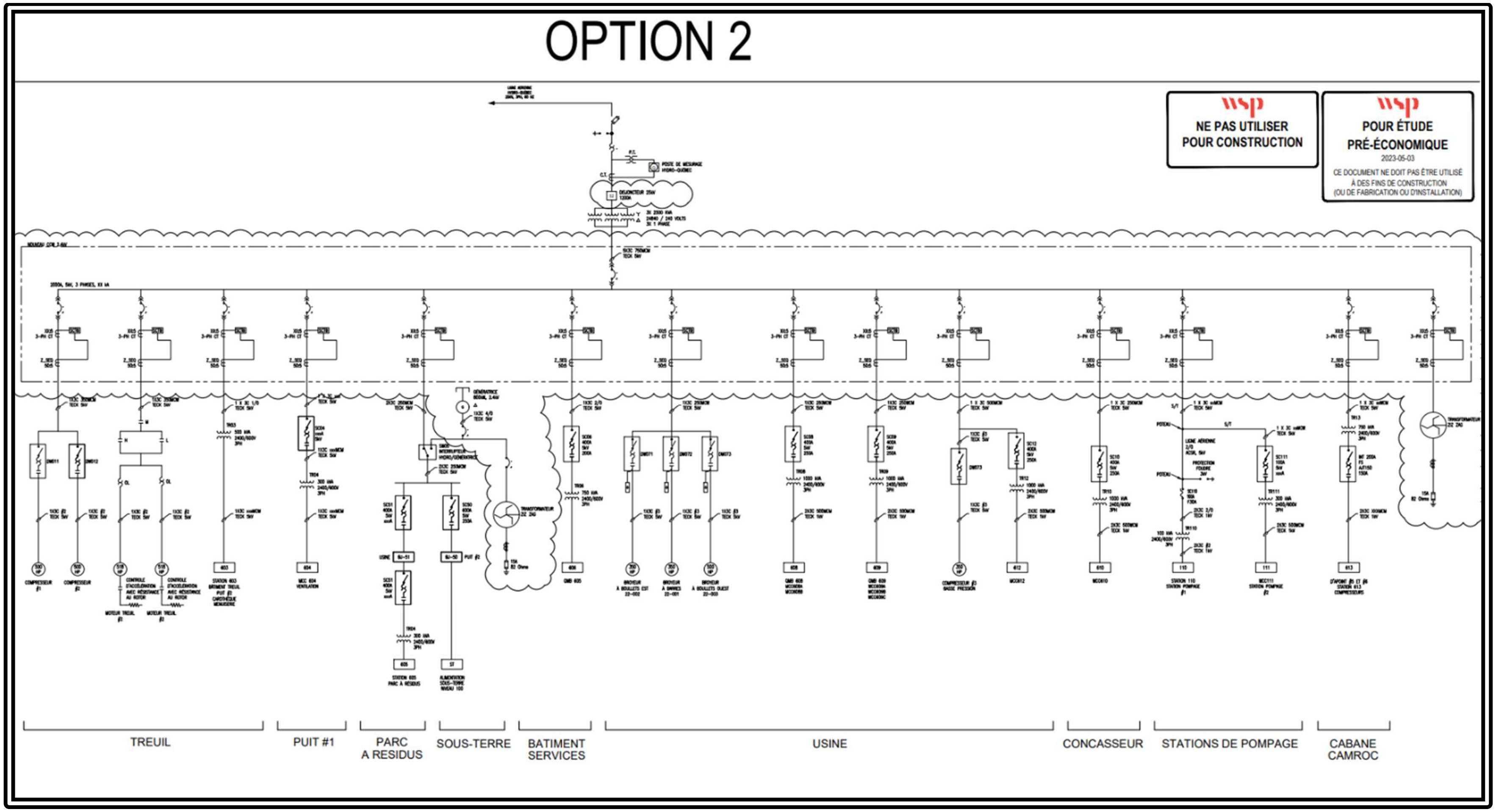

Figure 27.3 – Option 2 SLD – Drawing # 0000-E001-0201-0A

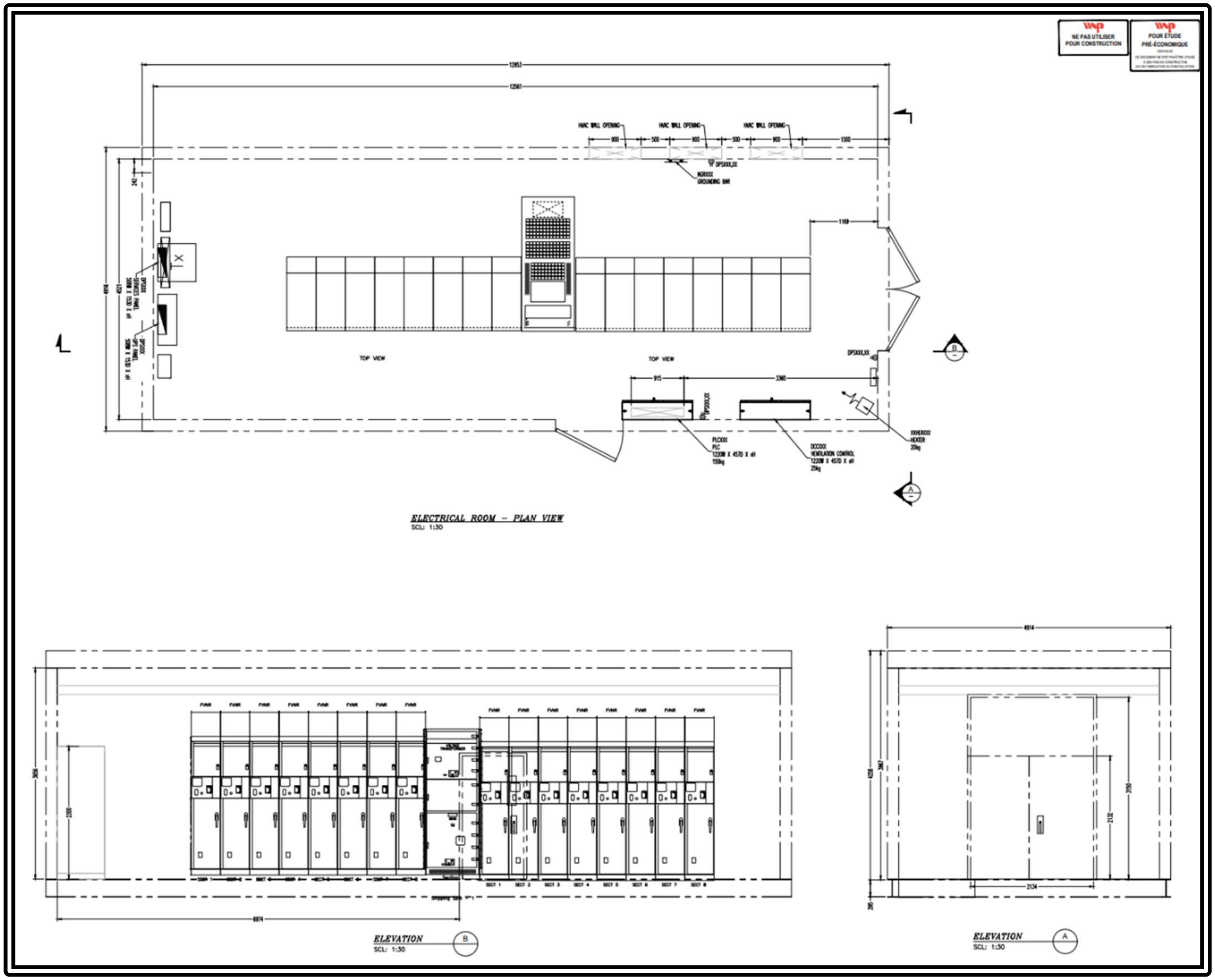

Figure 27.4 – Option 2 GA – Drawing # 0000-E001-0202-0A

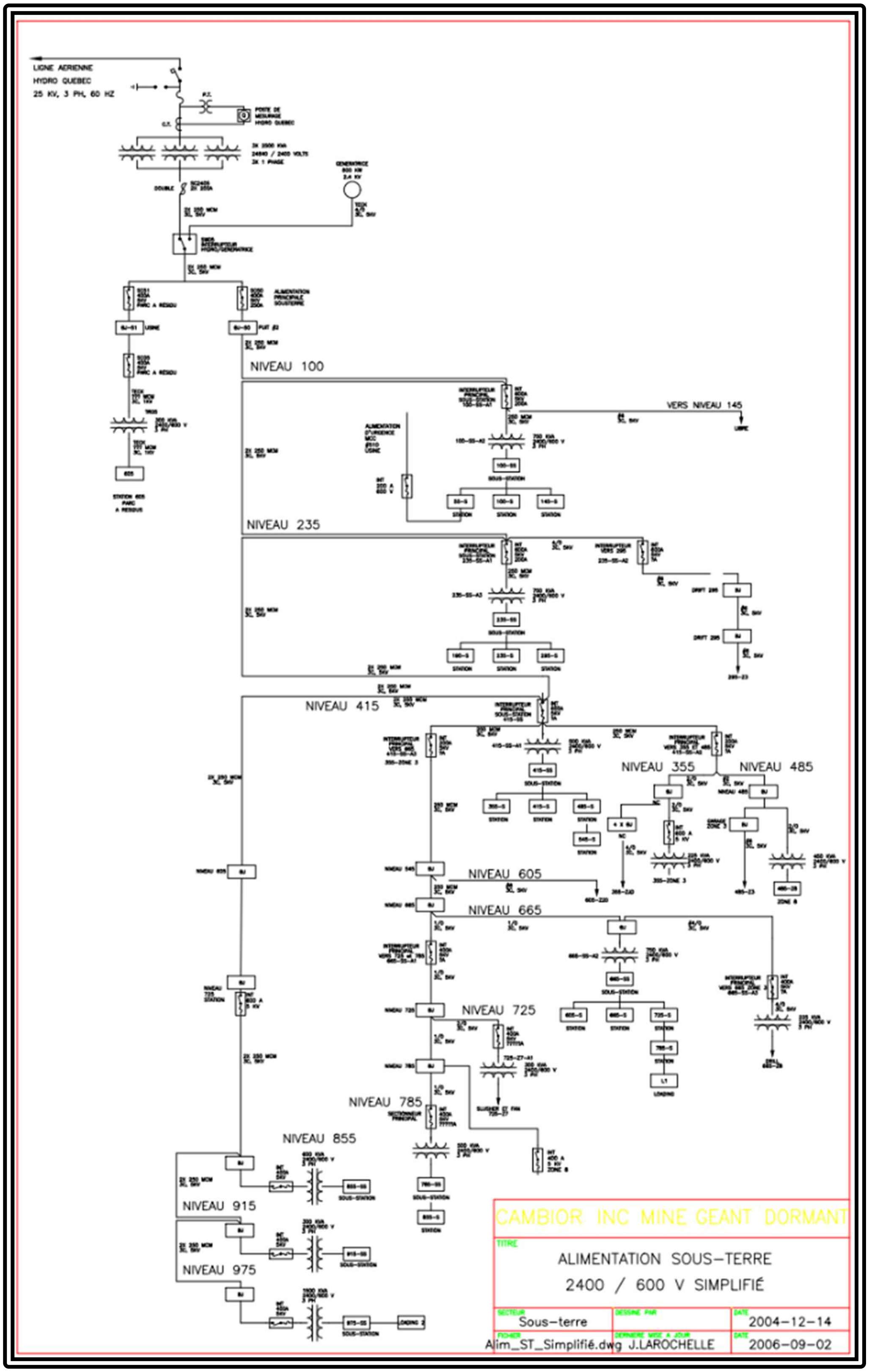

Figure 27.5 – SLD - UG simplified.

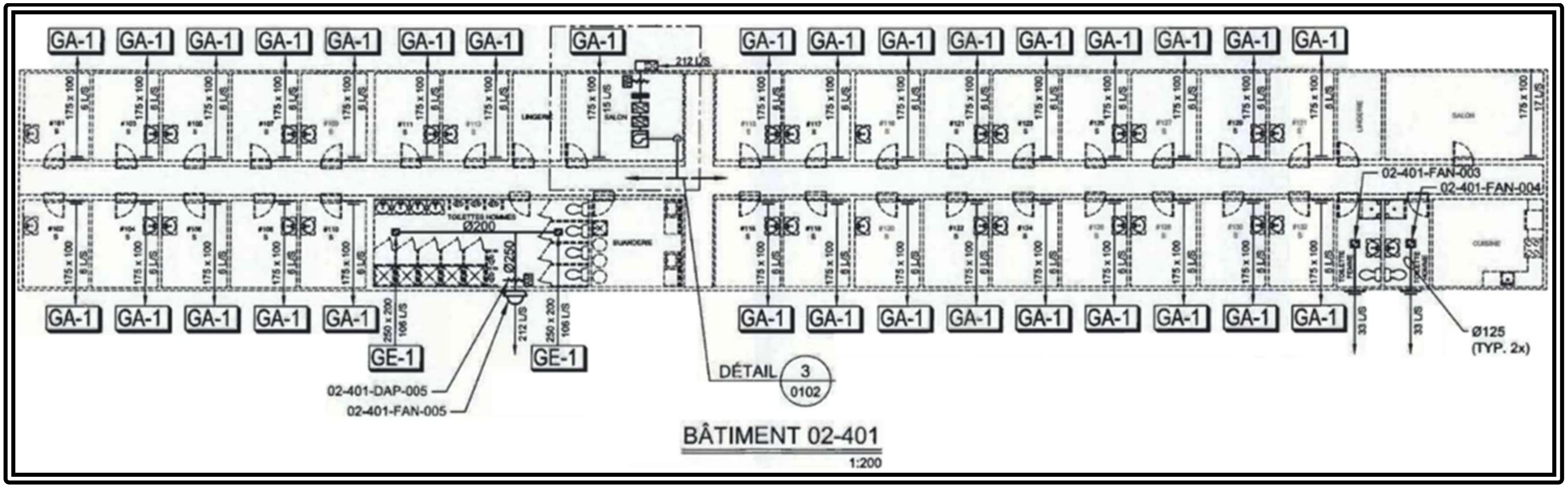

Figure 27.6 – Camp complex layout