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NI 43-101 Technical Report and Mineral Resource Estimate Update for the Duparquet Project, Quebec, Canada

Prepared for

First Mining Gold Corp.
Suite 2070 - 1188 West Georgia Street
V6E 4A2, Vancouver, BC, Canada

Project Location
Latitude: 48°30'34" North; Longitude: 79°12'34" West
Duparquet Township
Province of Quebec, Canada

Prepared by:

Marina Iund, P.Geo., M.Sc.
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Effective Date: September 12, 2022
Signature Date: October 06, 2022

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Effective Date: September 12, 2022

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NI 43-101 Technical Report and Mineral Resource Estimate Update for the Duparquet Project, Quebec, Canada

Prepared for

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Project Location
Latitude: 48°30'34" North; Longitude: 79°12'34" West
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CERTIFICATE OF AUTHOR – MARINA IUND

I, Marina Iund, P.Geo., M.Sc. (OGQ No. 1525, NAPEG No. L4431, PGO No. 3123), do hereby certify that:

1. I am employed as Senior Resources Geologist by InnovExplo Inc., located at 725, Boul. Lebourgneuf, Suite 312, Quebec City, Quebec, Canada, G2J 0C4.
1. This certificate applies to the report entitled "NI 43-101 Technical Report and Mineral Resource Estimate Update for the Duparquet Project, Quebec, Canada" (the "Technical Report") with an effective date of September 12, 2022, and signature date of October 06, 2022. The Technical Report was prepared for First Mining Gold Corp. (the "issuer").
2. I graduated with a Bachelor's degree in Geology from Université de Besançon (Besançon, France) in 2008. In addition, I obtained a Master's degree in Resources and Geodynamic from Université d'Orléans (Orléans, France), as well as a DESS degree in Exploration and Management of Non-renewable Resources from Université du Quebec à Montreal (Montreal, Quebec) in 2010.
3. I am a member of the Ordre des Géologues du Québec (OGQ No. 1525), the Association of Professional Geoscientists of Ontario (PGO, No. 3123), and the Northwest Territories and Nunavut Association of Professional Engineers and Professional Geoscientists (NAPEG licence No. L4431).
4. I have practiced my profession in mineral exploration, mine geology and resource geology for a total of 12 years since graduating from university. I acquired my expertise with Richmont Mines Inc. and Goldcorp. I have been a project geologist and then a senior geologist in mineral resources estimation for InnovExplo Inc. since September 2018.
5. I have read the definition of a qualified person ("QP") set out in Regulation 43-101/National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a QP for the purposes of NI 43-101.
6. I visited the Project on October 20, 2021, for the purpose of the Technical Report.
7. I am responsible for the overall supervision of the Technical Report. I am the principal author of and am responsible for items 2 to 12, 23 and 27. I am the co-author of and share responsibility for items 1, 14 and 25 to 26.
8. I am independent of the issuer applying all the tests in section 1.5 of NI 43-101.
9. I have not had prior involvement with the Project that is the subject of the Technical Report.
10. I have read NI 43-101, and the items of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
11. 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 06th day of October 2022 in Quebec City, Quebec, Canada.

(Original signed and sealed)

Marina Iund, P.Geo. (OGQ No. 1525), M.Sc.

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CERTIFICATE OF AUTHOR – CARL PELLETIER

I, Carl Pelletier, P.Geo. (OGQ No. 384, PGO No. 1713, EGBC No. 43167 and NAPEG No. L4160), do hereby certify that:

1. I am a professional geoscientist and Co-President Founder of InnovExplo Inc., located at 560, 3e Avenue, Val-d'Or, QC, Canada, J9P 1S4.
2. This certificate applies to the report entitled "NI 43-101 Technical Report and Mineral Resource Estimate Update for the Duparquet Project, Quebec, Canada" (the "Technical Report") with an effective date of September 12, 2022, and signature date of October 06, 2022. The Technical Report was prepared for First Mining Gold Corp. (the "issuer").
3. I graduated with a Bachelor's degree in Geology (B.Sc.) from Université du Québec à Montréal (Montreal, Quebec) in 1992. I initiated a Master's degree at the same university for which I completed the course program but not the thesis.
4. I am a member of the Ordre des Géologues du Québec (OGQ, No. 384), the Association of Professional Geoscientists of Ontario (PGO, No. 1713), the Association of Professional Engineers and Geoscientists of British Columbia (EGBC, No. 43167), the Northwest Territories Association of Professional Engineers and Geoscientists (NAPEG, No. L4160), and the Canadian Institute of Mines (CIM).
5. My relevant experience includes a total of 29 years since my graduation from university. My mining expertise was acquired at the Silidor, Sleeping Giant, Bousquet II, Sigma-Lamaque and Beaufor mines. My exploration experience was acquired with Cambior Inc. and McWatters Mining Inc. I have been a consulting geologist for InnovExplo Inc. since February 2004.
6. I have read the definition of a qualified person ("QP") set out in Regulation 43-101/National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a QP for the purposes of NI 43-101.
7. I visited the Project on November 16, 2011, and on February 7, 2012, both times for the purpose of the Technical Report.
8. I am the co-author of and share responsibility for items 1, 14, 25 and 26.
9. I am independent of the issuer applying all the tests in section 1.5 of NI 43-101.
10. I had prior involvement with the Project that is the subject of the Technical Report. I was QP for the NI43-101 Technical report entitled "Technical Report and Mineral Resource Estimate for the Duparquet Project" (August 2, 2013) and for the NI43-101 Technical report entitled "Technical Report and Prefeasibility study for the Duparquet Project" (May 23, 2014).
11. I have read NI 43-101, 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 06th day of October 2022 in Val-d'Or, Quebec, Canada.

(Original signed and sealed)

Carl Pelletier, P.Geo. (OGQ No. 384)

InnovExplo Inc.

carl.pelletier@innovexplo.com

CERTIFICATE OF AUTHOR – SIMON BOUDREAU

I, Simon Boudreau, P.Eng. (OIQ No. 132 338), do hereby certify that:

13. I am employed as Senior Mine Engineer by InnovExplo Inc., located at 560, 3e Avenue, Val-d'Or, Quebec, Canada, J9P 1S4.
14. This certificate applies to the report entitled "NI 43 101 Technical Report and Mineral Resource Estimate Update for the Duparquet Project, Quebec, Canada" (the "Technical Report") with an effective date of September 12, 2022, and signature date of October 06, 2022. The Technical Report was prepared for First Mining Gold Corp. (the "issuer").
15. I graduated with a Bachelor's degree in mining engineering (B.Ing.) from Université Laval (Québec, Québec) in 2003.
16. I am a member in good standing of the Ordre des Ingénieurs du Québec (No:132 338).
17. My relevant experience includes a total of nineteen (19) years since my graduation from university. I have been involved in mine engineering and production at Troilus mine for four (4) years, HRG Taparko mine for four (4) years, Dumas Contracting for three (3) years. I have also worked as independent consultant for the mining industry for five (5) years and with InnovExplo for three (3) years. As consultant I have been involved in many base metals and gold mining projects.
18. I have read the definition of a qualified person ("QP") set out in Regulation 43-101/National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a QP for the purposes of NI 43-101.
19. I have not visited the property for the purpose of the Technical Report.
20. I am the co-author of items 1 to 2 and 14, for which I share responsibility.
21. I am independent of the issuer applying all the tests in section 1.5 of NI 43-101.
22. I have not had prior involvement with the property that is the subject of the Technical Report.
23. I have read NI 43-101, and the items of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
24. 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 06th day of October 2022 in Trois-Rivières, Quebec, Canada.

(Original signed and sealed)

Simon Boudreau, P.Eng.
InnovExplo Inc.
Simon.boudreau@innovexplo.com

CERTIFICATE OF AUTHOR – GUY COMEAU

I, Guy Comeau, P.Eng. (OIQ No. 106546), do hereby certify that:

1. I am employed by Soutex at Suite 204, 1990 rue Cyrille-Duquet, Quebec City, QC G1N 4K8.
2. This certificate applies to the report entitled "NI 43-101 Technical Report and Mineral Resource Estimate Update for the Duparquet Project, Quebec, Canada" (the "Technical Report") with an effective date of September 12, 2022, and signature date of October 06, 2022. The Technical Report was prepared for First Mining Gold Corp. (the "issuer").
3. I graduated with a Bachelor's degree in Metallurgical Engineering from Technical University of Nova Scotia (Halifax, Nova Scotia) in 1990.
4. I am a member of the Ordre des Ingénieurs du Québec (OIQ, No. 106546).
5. I have worked as a metallurgical engineer for a total of 29 years since graduating from university. My expertise was acquired while working in various capacities, summarized as follows:
    a. Extensive experience in concentrator operations for various commodities including nickel, copper, zinc, lithium, and gold.
    b. Chief metallurgist at two nickel-copper operations in Quebec.
    c. Metallurgist at a copper operation in Quebec.
    d. Metallurgist at a copper, lead, zinc operation in New Brunswick.
    e. Senior consultant on pre-feasibility and feasibility studies on various commodities, including iron, nickel, copper and gold.
    f. Senior consultant providing onsite assistance for lithium, nickel-copper, gold and iron concentrators.
6. I have read the definition of a qualified person ("QP") set out in Regulation 43-101/National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a QP for the purposes of NI 43-101.
7. I have not visited the Project for the purpose of the Technical Report.
8. I am responsible for item 13 and share responsibility for items 1, 25 and 26. For sections 13.1.7, 13.1.4.2, 13.1.5.2, 13.1.8.2, part of 13.1.2.2 and 13.1.10, I relied solely on the information contained in the previous NI 43-101 technical report issued in March 2014.
9. I am independent of the issuer applying all the tests in section 1.5 of NI 43-101.
10. I have not had prior involvement with the Project that is the subject of the Technical Report.
11. I have read NI 43-101 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 06th day of October 2022 in North Tetagouche, New Brunswick, Canada.

(Original signed and sealed)

Guy Comeau, P.Eng., OIQ No. 106546

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1. SUMMARY

Introduction

First Mining Gold Corp. (“First Mining” or the “issuer”) retained InnovExplo Inc. (“InnovExplo”) to prepare a Technical Report (the “Technical Report”) to present and support the results of an updated Mineral Resource Estimate (the “2022 MRE”) for the Duparquet Project (the “Project”), located in the province of Quebec, Canada.

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

First Mining is a gold project development company listed on the Toronto Stock Exchange (“TSX”) under the symbol ‘FF’. Its head office and exploration office are at Suite 2070, 1188 West Georgia Street, Vancouver, British Columbia, Canada, V6E 4A2.

InnovExplo is an independent mining and exploration consulting firm based in Val-d’Or, Quebec, Canada.

Soutex is an independent metallurgical consulting firm based in Quebec City, Quebec, Canada.

The 2022 MRE follows the 2014 CIM Definition Standards on Mineral Resources and Mineral Reserves (“CIM Definition Standards”) and the 2019 CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (“CIM Guidelines”).

The Duparquet Project consists of four (4) contiguous properties: Beattie, Donchester, Central Duparquet and Dumico. The Project comprises the mineral exploration properties and the mine tailings from the former Beattie mine, a portion of which extends beyond the Project boundaries.

Property Description, Location and Access

The Project is located in the province of Quebec, Canada, just north of the town of Duparquet. It lies in Duparquet Township, in the Abitibi-Ouest regional county municipality (“RCM”), which is part of the Abitibi-Témiscamingue administrative region.

The Project can be reached by paved, two-lane, all-season provincial highways from Rouyn-Noranda (53 km to the south; Route 393 and Route 101) or La Sarre (33 km to the north; Route 393). The highways pass through parts of the Project and several gravel roads also lead onto it.

The Project comprises fifty (50) map-designated claims (“CDC”) covering an area of 1,079.2 ha.

Geology

The Project is located in the southern portion of the Superior Province, within the Abitibi terrane. The Project straddles syenitic plutons and the Kinojevis, Duparquet and Mont-Brun formation. The Project area is characterized by the presence of two syenitic plutons oriented E-W. These intrusions are bounded by E-W major faults, which are interpreted as splays of the main SE-trending Destor-Porcupine-Manneville Fault Zone (“DPMFZ”), which clips the southwest corner of the Project. The geological formations generally strike

E-W and dip steeply (80°-85°) to the north. The metamorphic grade is low (greenschist facies), and local alteration is represented by chloritization, silicification and sericitization.

The rocks underlying the Project are generally made up of intercalated felsic and mafic metavolcanic flows, with the felsic flows being the oldest. Metasedimentary layers are also present and are generally more prevalent on the south side of the DPMFZ. All the units have been intruded by syenite porphyry units, which appear to be concordant with the location of the major fault zones. Quartz feldspar and lath porphyries were injected along minor faults affecting the syenite intrusions.

The predominant structures on the Project are the E-W splays of the DPMFZ. The Beattie Fault Zone ("BFZ") is located along the north contact of the main syenite body, whereas the Donchester Fault Zone ("DFZ") is located along the south contact. The Central Duparquet Fault Zone ("CDFZ") is located along the south contact of the second smaller syenite intrusive to the east. The syenite porphyry generally plunges to the east.

Mineralization

According to Bevan (2011), the "main" type of gold mineralization in the Duparquet deposit generally occurs within shears or brecciated zones along or within the adjacent intrusive syenitic masses and is associated with finely disseminated pyrite and minor arsenopyrite replacement. Sulphide content is generally low (0.5 to 4%), although it can be up to 10% in some cases. Higher gold grades appear to be related to the finer-grained sulphides. Historically, gold production at the Beattie mine was accompanied by the extraction of arsenic trioxide and silver as by-products. The "breccia" type of mineralized material is found within the metavolcanic rocks (volcanics and tuffs) and consists of well-mineralized, siliceous, brecciated, grey-coloured and bleached zones. The porphyry-type mineralized material consists of fine-grained and strongly silicified mineralized zones hosted by porphyry intrusives.

The typical mineral assemblage in mineralized zones of all types is characterized by feldspar, quartz, sulphides (pyrite and arsenopyrite), sericite, chlorite and other secondary minerals. Mill tests suggest that some 35% of the gold is in a free state, with the remainder associated with sulphides (Bevan, 2011).

Deposit type

The standard orogenic gold model characterizes the majority of gold deposits within the Abitibi belt. However, several examples of late mineralization are associated with alkaline intrusions (Robert, 2001), thus differing from the standard orogenic gold model. Syenite-associated disseminated gold deposits in the Abitibi greenstone belt consist of zones of disseminated sulphides with variably developed quartz stockworks, which are intimately associated with Timiskaming-age, monzonitic to syenitic porphyry intrusions (Robert, 2001). Like quartz-carbonate vein deposits, all known syenite-associated disseminated gold deposits in the southern Abitibi belt occur along a major fault. Examples of these deposits are Young-Davidson, Matachewan Consolidated, Ross, Holt-McDermott, and Lightening in Ontario; and Beattie, Douay, Canadian Malartic, East Malartic, and Barnat-Sladen in Quebec.

Syenite-associated disseminated gold deposits consist of zones of disseminated sulphides with variably developed stockworks in intensely altered wallrocks (Robert, 2001). Owing to the abundance of micro-veinlet stockworking and fracturing, many orebodies take on a breccia appearance. Syenitic intrusions indirectly control the location of gold deposits by their effect on the development of potentially mineralized structures during diapirism.

Gold deposits in the Duparquet area are genetically related to these intrusive rocks and tend to occur close to the syenite-sedimentary and/or volcanic rock contacts. This is attributed to the competency contrasts between the syenitic rocks and the Duparquet Formation lithologies during deformation, resulting in favourable structural traps for gold mineralization. The main gold mineralization is associated with a network of E-W dextral strike-slip faults dipping steeply to the north.

In the Duparquet area, the main alteration observed is silicification. Sericitization, ankeritization and chloritization are other types of alteration associated with gold mineralization (Goutier and Lacroix, 1992). The main sulphide is pyrite (<10%), accompanied by arsenopyrite (Davidson and Banfield, 1944). Gold is hosted in arsenopyrite and arsenian pyrite (Bigot and Jébrack, 2012).

Mineral Resource Estimates

The mineral resource estimate update for the Project (the “2022 MRE”) was prepared by Marina Iund, P.Geo., Carl Pelletier, P.Geo., Simon Boudreau, P.Eng. and Guy Comeau, P.Eng. using all available information. The main objective was to update the results of InnovExplo’s previous mineral resource estimate for the Project, dated June 26, 2013 (Poirier et al., 2014). The updated estimate includes new drill holes on the Beattie, Donchester and Central Duparquet properties.

The authors have classified the current mineral resource estimate as measured, indicated and inferred resources based on data density, search ellipse criteria, drill hole spacing and interpolation parameters. The authors also believe that the requirement of “reasonable prospects for eventual economic extraction” has been met by having:

- Resources constrained by a pit shell, with a 50° angle in rock and a 30° angle in overburden;
- Constraining volumes applied to any blocks (potential underground extraction scenario) using the Deswik Stope Optimizer (DSO) for the out-pit resources; and
- Cut-off grades based on reasonable inputs amenable to potential open-pit and underground extraction scenarios.

The 2022 MRE is considered reliable and based on quality data and geological knowledge. The estimate follows CIM Definition Standards.

The following table presents the 2022 Mineral Resource Estimate for the Global Duparquet Project broke down by mining method, at the actual cut off grade.

INNOVEXPLO

2022 Mineral Resource Estimate for the Global Duparquet Project, by mining method (Table 14-14)

Area (Mining Method)	Cut-off (g/t)	Measured resource			Indicated resource			Inferred resource		
		Tonnage (t)	Au (g/t)	Ounces	Tonnage (t)	Au (g/t)	Ounces	Tonnage (t)	Au (g/t)	Ounces
Open pit	0.4	163,700	1.37	7,200	59,410,600	1.52	2,909,600	28,333,000	1.07	970,400
UG mining	1.5	-	-	-	5,506,900	2.26	399,300	9,038,900	2.29	665,600
Tailings	0.4	19,900	2.03	1,300	4,105,200	0.93	123,200	-	-	-
Total	-	183,600	1.43	8,500	69,022,700	1.55	3,432,100	37,371,900	1.36	1,636,000

Notes to accompany the Mineral Resource Estimate:

1. The independent and qualified persons for the mineral resource estimate, as defined by NI 43-101, are Marina Iund, P.Geo., Carl Pelletier, P.Geo., Simon Boudreau, P.Eng., all from Innovexplo and Guy Comeau, P.Eng. from Soutex. The effective date of the estimate is September 12, 2022.
2. These mineral resources are not mineral reserves, as they do not have demonstrated economic viability. There is currently insufficient data to define these Inferred mineral resources as Indicated or Measured mineral resources and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured mineral resource category. The mineral resource estimate follows current CIM Definition Standards.
3. The results are presented in situ and undiluted and have reasonable prospects of economic viability.
4. In-pit and Underground estimates encompass sixty (60) mineralized domains and one dilution envelop using the grade of the adjacent material when assayed or a value of zero when not assayed; The tailings estimate encompass four (4) zones.
5. In-pit and Underground: High-grade capping of 25 g/t Au; Tailings: High-grade capping of 13.0 g/t Au for Zone 1, 3.5 g/t Au for Zone 2, 1.7 g/t Au for Zone 3 and 2.2 g/t Au for Zone 4. High-grade capping supported by statistical analysis was done on raw assay data before compositing.
6. In-pit and Underground: The estimate used a sub-block model in GEOVIA SURPAC 2021 with a unit block size of 5m x 5m x 5m and a minimum block size of 1.25m x 1.25m x1.25m. Grade interpolation was obtained by ID2 using hard boundaries. Tailings: The estimate used a block model in GEOVIA GEMS with a block size of 5m x 5m x 1m. Grade interpolation was obtained by ID2 using hard boundaries.
7. In-pit and Underground: A density value of 2.73 g/cm^3 was used for the mineralized domains and the envelope. A density value of 2.00 g/cm^3 was used for the overburden. A density value of 1.00 g/cm^3 was used for the excavation solids (drifts and stopes) assumed to be filled with water. Tailings: A fixed density of 1.45 g/cm^3 was used in zones and waste.
8. In-pit and Underground: The mineral resource estimate is classified as Measured, Indicated and Inferred. The measured category is defined by blocks having a volume of at least 25% within an envelope built at a distance of 10 m around existing channel samples. The Indicated category is defined by blocks meeting at least one (1) of the following conditions: Blocks falling within a 15-m buffer surrounding existing stopes and/or blocks for which the average distance to composites is less than 45 m. A clipping polygon was generated to constrain Indicated resources for each of the sixty (60) mineralized domains. Only the blocks for which reasonable geological and grade continuity have been demonstrated were selected. All remaining interpolated blocks were classified as Inferred resources. Blocks interpolated in the envelope were all classified as Inferred resources. Tailings: The Measured and Indicated categories were defined based on the drill hole spacing (Measured: Zones 1 and 2 = 30m x 30m grid; Indicated: Zone 3 = 100m x 100m grid and Zone 4 = 200m x 200m grid).
9. In-pit and Underground: The mineral resource estimate is locally pit-constrained with a bedrock slope angle of 50° and an overburden slope angle of 30°. The out-pit mineral resource met the reasonable prospect for eventual economic extraction by having constraining volumes applied to any blocks (potential underground extraction scenario) using DSO. It is reported at a rounded cut-off grade of 0.4 g/t Au (in-pit and tailings) and 1.5 g/t Au (UG). The cut-off grades were calculated using the following parameters: mining cost = CA$70.00 (UG); processing cost = CA$11.9 to 17.0; G&A = CA$8.75; refining and selling costs = CA$ 5.00; gold price = US$ 1,650/oz; USD:CAD exchange rate = 1.31; and mill recovery = 93.9%. The cut-off grades should be re-evaluated in light of future prevailing market conditions (metal prices, exchange rates, mining costs etc.).
10. The number of metric tons and ounces was rounded to the nearest hundred, following the recommendations in NI 43-101. Any discrepancies in the totals are due to rounding effects.
11. The authors are not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, or marketing issues, or any other relevant issue not reported in the Technical Report, that could materially affect the Mineral Resource Estimate.

The in-pit and underground portion of the 2022 MRE represents an increase of 11% in the Indicated Resource ounces and an increase of 13% in the inferred Resource ounces compared to the previous 2014 MRE (Poirier et al., 2014). Measured Resource ounces decreased by 6.3% mainly due to the use of constraining volumes. This increase is due to the addition of the 55 new assayed holes that were drilled on the Project since 2013 and the adjustment of the economic parameters to reflect current economic conditions.

Interpretation and Conclusions

The authors conclude the following:

- The database supporting the 2022 MRE is complete, valid and up to date.
- Geological and gold-grade continuity has been demonstrated for all 72 mineralized zones.
- The key parameters of the 2022 MRE (density, capping, compositing, interpolation, search ellipsoid, etc.) are supported by data and statistical and/or geostatistical analysis.
- The 2022 MRE includes measured, indicated and inferred resources for a combination of two mining scenarios: open pit and selective underground. The 2022 MRE complies with CIM Definition Standards and CIM Guidelines.
- Two cut-off grades of 0.40 and 1.50 g/t Au were used, corresponding to potential open pit and selective underground mining scenarios.
- Cut-off grades were calculated at a gold price of US$1,650 per troy ounce and an exchange rate of 1.31 USD/CAD, using reasonable mining, processing and G&A costs.
- In a combined pit and selective underground mining scenario, the Project contains an estimated M+I Resource of 65,081,200 t at 1.58 g/t Au for 3,316,100 oz of gold and an Inferred Resource of 37,371,900 t at 1.36 g/t Au for 1,636,000 oz of gold. The Project also contains the Beattie mine tailings with an estimated M+I Resource of 4,125,100 t at 0.94 g/t Au for 124,500 oz of gold.
- The results of the 2022 MRE represent a 10.5% increase in the M+I Resource and a 13.4% increase in the Inferred Resource compared to the previous 2014 MRE of Poirier et al., 2014 (Table 25-1). The increase in the M+I Resource is due to a deeper optimized shell and the updated economic parameters. The same reasons combined with the addition of 55 drill holes explain the increase in Inferred resources.
- Based on metallurgical tests, the Duparquet project appears amenable to existing gold recovery processes. A combination of flotation, pressure oxidation and cyanide leach processes has shown a gold recovery ranging from 94.7% to 96.5%.
- Additional diamond drilling on multiple zones would likely upgrade some of the Inferred Resource to the Indicated category and/or add to the Inferred Resource since most of the mineralized zones have not been fully explored at depth or close to surface infrastructures.

At this stage, it is reasonable to believe that a hybrid operation consisting of an early open pit followed by later underground mining activities is amenable to the expectation of “reasonable prospects of eventual economic extraction” as stated in the CIM Guidelines. The potential to add new resources in the open pit through exploration is best focused in the east direction because the favorable geology hosting the Project mineralization is constrained to the west, and the pit itself is constrained to the south by the Project limit and the town of Duparquet. Drilling to tighten the drill spacing in the inferred resources will permit a transfer from inferred to indicated by adding confidence in the estimate. There is potential to add material at depth below the existing mineralized model that could be accessed from the underground infrastructures. The reader is cautioned that this exploration target is not a mineral resource estimate and is conceptual in nature. There has been insufficient exploration to define it as a mineral resource and it is uncertain if further exploration will delineate the exploration target as a mineral resource.

The authors consider the 2022 MRE reliable, thorough, and based on quality data, reasonable hypotheses, and parameters compliant with NI 43-101 requirements, CIM Definition Standards and CIM Guidelines.

Recommendations

Geology

The authors recommend further exploration drilling on the Project to potentially increase resources and the confidence level of the geological model. More specifically, the NE-SW striking secondary zones should be drilled to test their lateral and depth extensions. Additional detailed surface mapping and channel sampling would enhance the structural model. A compilation of historical diamond drill holes on the Central Duparquet Property is recommended, particularly those drilled during the 1980s to cover the current mineralized zones in that area. Further definition drilling is recommended along strike and at depth to upgrade the Inferred resources to the Indicated category and address the underground potential for all zones.

Mining

Additional geotechnical and hydrogeological studies should be undertaken to better define the pit wall slopes presented in this report. This would involve confirming the structural data over the proposed footprint of the open pit. Ideally, this would involve a geotechnical drilling program with a minimum of one (1) hole oriented perpendicular to each of the four pit walls (north, south, east and west).

Metallurgy

The QP recommends the following tests:

- Additional variability hardness tests throughout the Project. Several samples from each zone should undergo SMC, rod mill and ball mill Bond tests.
- A variability locked-cycle testing program for the existing tailings area involving a mix of tailings and ore.
- The metallurgical results using the POX process have been positive. However, the QP believes that the BIOX and Albion processes should be re-investigated.

Costs Estimate for Recommended Work

The authors have prepared a cost estimate for the recommended two-phase work program to serve as a guideline. The budget for the proposed program is presented in the following table. Expenditures for Phase 1 are estimated at C$4.65 million (incl. 15% for contingencies). Expenditures for Phase 2 are estimated at C$1.25 million (incl. 15%

for contingencies). Phase 2 is contingent upon the success of Phase 1. The grand total is C$5,9 million (incl. 15% for contingencies).

Estimated Costs for the Recommended Work Program (Table 26-1)

Phase 1	Work program	Budget cost
a	Drilling and Compilation of historical drill holes	$2.5 M
b	Resource Update	$150 k
c	Geotechnical Drilling	$1.0 M
d	Metallurgical testwork	$1.0 M
	Phase 1 subtotal	$4.65 M
Phase 2	**Work program**	**Budget cost**
a	Environmental Assessment	$800 K
b	Hydrogeological Study	$250 K
	Phase 2 subtotal	$1.05 M
	TOTAL (Phase 1 and Phase 2)	$5.7 M

2. INTRODUCTION

2.1 Overview of Terms of Reference

First Mining Gold Corp. ("First Mining" or the "issuer") retained InnovExplo Inc. ("InnovExplo") to prepare a Technical Report (the "Technical Report") to present and support the results of an updated Mineral Resource Estimate (the "2022 MRE") for the Duparquet Project (the "Project"), located in the province of Quebec, Canada.

This Technical Report was prepared in accordance with Canadian Securities Administrators' National Instrument 43-101 Respecting Standards of Disclosure for Mineral Projects ("NI 43-101") and Form 43-101F1.

First Mining is a gold project development company listed on the Toronto Stock Exchange ("TSX") under the symbol 'FF'. Its head office and exploration office are at Suite 2070, 1188 West Georgia Street, Vancouver, British Columbia, Canada, V6E 4A2.

InnovExplo is an independent mining and exploration consulting firm based in Val-d'Or, Quebec, Canada.

Soutex is an independent metallurgical consulting firm based in Quebec City, Quebec, Canada.

The 2022 MRE follows the 2014 CIM Definition Standards on Mineral Resources and Mineral Reserves ("CIM Definition Standards") and the 2019 CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines ("CIM Guidelines").

The Duparquet Project consists of four (4) contiguous properties: Beattie, Donchester, Central Duparquet and Dumico. The Project comprises the mineral exploration properties and the mine tailings from the former Beattie mine, a portion of which extends beyond the Project boundaries.

2.2 Report Responsibility, Qualified Persons

This technical report was prepared by Marina Iund (P.Geo.), Senior Resources Geologist of InnovExplo, Carl Pelletier (P.Geo.), Co-President Founder of InnovExplo, Simon Boudreau (P.Eng.), Senior Mine Engineer of Innovexplo and Guy Comeau (P.Eng.) Senior Metallurgist of Soutex Inc. All four authors are qualified persons ("QPs") as set out in NI 43-101.

Ms. Iund is a professional geologist in good standing with the OGQ (licence No. 1525), PGO (licence No. 3123) and NAPEG (licence No. L4431). She is responsible for the overall supervision of the Technical Report. She is the principal author of and responsible for items 2 to 12, 23 and 27, and the co-author of and shares responsibility for items 1, 14 and 25 to 26.

Mr. Pelletier is a professional geologist in good standing with the OGQ (licence No. 384), PGO (licence No. 1713), EGBC (licence No. 43167) and NAPEG (No. L4160). He is the co-author of and shares responsibility for items 1, 14, 25 and 26.

Mr. Boudreau is a professional engineer in good standing with the OIQ (permit No. 1320338). He is the co-author of sections 1 to 2 and 14, for which he shares responsibility.

Mr. Comeau is a professional engineer in good standing with the OIQ (licence No. 106546). He is responsible for item 13 and share responsibility for items 1, 25 and 26.

2.3 Site Visits

Ms. Iund visited the Project on October 20, 2021. During the visit, she reviewed selected drill core intervals and inspected the core storage facility. She also surveyed drill hole collars for independent validation.

Mr. Pelletier visited the Project on November 16, 2011, and on February 7, 2012. His Project tour included a general visual inspection of buildings, the local roads, historical tailings and flooded underground openings connected to the surface. He also reviewed selected drill core intervals, inspected the core storage facility and surveyed selected drill hole collars for independent validation.

2.4 Effective Date

The close-out date of the mineral resource database is October 20, 2021.

The effective date of the 2022 MRE is September 12, 2022.

2.5 Sources of Information

The information described in Item 3 and the documents listed in Item 27 were used to support this Technical Report. Excerpts or summaries from documents authored by other consultants are indicated in the text.

The authors' assessment of the Project was based on published material in addition to the data, professional opinions and unpublished material submitted by the issuer. The authors reviewed all relevant data provided by the issuer and/or by its agents.

The author also consulted other sources of information, mainly the Government of Quebec's online databases for mining title management and assessment work (GESTIM and SIGEOM, respectively), as well as the issuer's filings on SEDAR namely annual information forms, MD&A reports, press releases and previous technical reports.

The authors reviewed and appraised all the information used to prepare this Technical Report and believe that such information is valid and appropriate considering the status of the project and the purpose for which this Technical Report is prepared. The authors have thoroughly researched and documented the conclusions and recommendations herein.

2.6 Currency, Units of Measure, and Acronyms

The abbreviations, acronyms and units in this Technical Report are provided in Table 2-1 and Table 2-2. All currency amounts are expressed 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 base metal 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 2-1 – List of abbreviations and acronyms

Abbreviation or acronym	Term
3DL	Three times the detection limit
3SD	Three times standard deviations
43-101	National Instrument 43-101 (Regulation 43-101 in Quebec)
AAS	Atomic absorption spectroscopy
ABA	Acid base accounting
Ai	Abrasion index
AP	Potential acid
BFS	Basic iron sulphate
BFZ	Beattie Fault Zone
BWi	Bond work index
CAD:USD	Canadian-American exchange rate
CAPEX	Capital expenditure
CCD	Counter current decantation
CDFZ	Central Duparquet Fault Zone
CIL	Carbon in leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIM Definition Standards	CIM Definition Standards for Mineral Resources and Mineral Reserves (2014)
CIM Guidelines	CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (2019)
CLLFZ	Cadillac–Larder Lake Fault Zone
CNWAD	Cyanide weak acid dissociable
CoG	cut-off grade
CRM	Certified reference material
CND	Cyanide destruction
CNT	Total cyanide
COV	Coefficient of variation
CSD	Critical solid density
CSFA	Conglomerate-sandstone facies association
CVAAS	Vapour atomic absorption spectroscopy
CWi	Crusher work index
DEM	Digital elevation model
DDH	Diamond drill hole

Abbreviation or acronym	Term
DFZ	Donchester Fault Zone
DPMFZ	Destor-Porcupine-Manneville fault zone
DSO	Deswik Stope Optimizer
EA	Environmental assessment
EGBC	Association of professional engineers and geoscientists of British Columbia
F_{80}	80% passing - Feed
FS	Feasibility study
G&A	General and administration
GESTIM	Gestion des titres miniers (the MERN's online claim management system)
HPGR	High Pressure Grinding Rolls
ICP-OES	Inductively coupled plasma - optical emission spectrometry
ID2	Inverse distance squared
IEC	International Electrotechnical Commission
ISO	International Organization for Standardization
ISR	Initial settling rate
MD&A	Management discussion and analysis
M+I	Measured & Indicated
MERN	Ministère de l'Énergie et des Ressources Naturelles du Québec (Quebec's Ministry of Energy and Natural Resources)
MRC	Municipalité régionale de comté (Regional county municipality in English)
MRE	Mineral resource estimate
NAD	North American Datum
NAD83	North American Datum of 1983
NAG	Net acid generation
NAPEG	Association of professional engineers and professional geoscientists
NI 43-101	National Instrument 43-101 (Regulation 43-101 in Quebec)
NN	Nearest neighbour
NP	Neutralization potential
NSR	Net smelter return
NTS	National topographic system
NVZ	Northern volcanic zones
OGQ	Ordre des Géologues du Québec
OIQ	Ordre des Ingénieurs du Québec
OK	Ordinary kriging
OPEX	Operational expenditure
P_{80}	80% passing - Product
PAG	Potentially acid generating

Abbreviation or acronym	Term
PEA	Preliminary economic assessment
PFS	Prefeasibility study
PGO	Association of professional geoscientists of Ontario
POX	Pressure oxidation
P+P	Proven & Probable
QA	Quality assurance
QA/QC	Quality assurance/quality control
QC	Quality control
QP	Qualified person (as defined in National Instrument 43-101)
RCM	Regional county municipality (*Municipalité régionale de comté* or MRC in French)
Regulation 43-101	National Instrument 43-101 (name in Quebec)
RQD	Rock quality designation
RWi	Rod work index
SAFA	Sandstone-argillite facies association
SAG	Semi-autogenous-grinding
SCC	Standards Council of Canada
SD	Standard deviation
SEDAR	System for electronic document analysis and retrieval
SG	Specific gravity
SIGÉOM	Système d'information géominière (the MERN's online spatial reference geomining information system)
SMC	SAG mill comminution
SVZ	Southern volcanic zones
TCLP	Toxicity characteristic leaching procedure
TDEM	Time domain electromagnetic method
TDS	Total dissolved solids
THUA	Thickener hydraulic unit area
TSS	Supernatant total suspended solids
TSX	Toronto stock exchange
TUFUA	Thickener underflow unit area
UG	Underground
UTM	Universal Transverse Mercator coordinate system
XRF	X-ray Fluorescence
Wio	Operating work index

Table 2.2 – List of units

Symbol	Unit
%	Percent
% solids	Percent solids by weight
$, C, CA, CAD	Canadian dollar
$/t	Dollars per metric ton
°	Angular degree
°C	Degree Celsius
µm	Micron (micrometre)
A	Ampere
Axb	Resistance to impact
cm	Centimetre
cm^3	Cubic centimetre
d	Day (24 hours)
dm	Decametre
ft	Foot (12 inches)
g	Gram
G	Billion
Ga	Billion years
g/cm^3	Gram per cubic centimetre
g/d	Gram per day
g/g	Gram per gram
g/L	Gram per litre
g/t	Gram per metric ton (tonne)
h	Hour (60 minutes)
ha	Hectare
k	Thousand (000)
kg	Kilogram
kg/d	Kilogram per day
kg/h	Kilogram per hour
$kg/m^2/h$	Kilogram per square meter per hour
kg/t	Kilogram per metric ton
km	Kilometre
km^2	Square kilometre
kPa	Kilopascal
kV	Kilovolt
kW	Kilowatt
kWh	Kilowatt-hour

Symbol	Unit
kWh/t	Kilowatt-hour per metric ton
L	Litre
L/kg	Litre per kilogram
M	Million
m	Metre
m^2	Square metre
m^3	Cubic metre
m/d	Metre per day
m^3/h	Cubic metre per hour
$m^3/m^2/d$	Cubic metre per square metre per day
$m^3/m^2/h$	Cubic metre per square metre per hour
m^2/tpd	Square metre per metric tonne per day
Ma	Million years (annum)
masl	Metres above mean sea level
mn	Minute (60 seconds)
mm	Millimetre
mm^2	Square millimetre
Moz	Million (troy) ounces
Mt	Million metric tons
mV	Millivolt
MW	Megawatt
N/mm^2	Newton per square millimetre
oz	Troy ounce
oz/t	Ounce (troy) per short ton (2,000 lbs)
ppb	Parts per billion
ppm	Parts per million
psi	Pounds per square inch
rpm	Revolutions per minute
s	Second
s^2	Second squared
S	Sulphur
t	Metric tonne (1,000 kg)
t of O_2/t_{conc}	Metric tonne of O_2 per metric tonne of concentrate
ta	Abrasion breakage
ton	Short ton (2,000 lbs)
tpd	Metric tonne per day
t/h	Metric tonne per hour

Symbol	Unit
t/h.g	Metric tonne per hour times gramme
$t/m^2/h$	Metric tonne per square meter per hour
t/t	Metric tonne by metric tonne
US$	American dollar
wt%	Weight percent
w/w	Weight per weight
y	Year (365 days)

Table 2-2 – Conversion Factors for Measurements

Imperial Unit	Unit Multiplied by Metric Unit	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

InnovExplo has followed standard professional procedures in preparing the contents of this Technical Report. The report is based upon information believed to be accurate at the time of writing, considering the status of the Project and the purpose for which the report was prepared. The data have been verified where possible. InnovExplo has no reason to believe that the data were not collected in a professional manner.

The authors did not rely on other experts to prepare this Technical Report. It was prepared by InnovExplo at the request of the issuer. Marina Iund (P.Geo.) and Carl Pelletier (P.Geo.) are the QPs responsible for reviewing the technical documentation relevant to the Technical Report, preparing a mineral resource estimate for the Project, and recommending a work program.

InnovExplo has not verified the legal status of, or legal title to, any claims, nor the legality of any underlying agreements that may exist concerning the properties as described in Item 4 of this report. The QPs have relied on the issuer's information about mining titles, option agreements, royalty agreements, environmental liabilities, and permits. Neither the QPs nor InnovExplo are qualified to express any legal opinion concerning Project titles, current ownership or possible litigation.

InnovExplo consulted GESTIM and SIGEOM over the course of the mandate. The websites were most recently viewed in April, 2022:

- gestim.mines.gouv.qc.ca/MRN_GestimP_Presentation/ODM02101_login.aspx
- sigeom.mines.gouv.qc.ca/signet/classes/I1102_indexAccueil?l=a

4. PROJECT DESCRIPTION AND LOCATION

4.1 Location

The Project is located in the province of Quebec, Canada, just north of the town of Duparquet and 50 km north of the city of Rouyn-Noranda. It lies in Duparquet Township, in the Abitibi-Ouest regional county municipality (“RCM”), which is part of the Abitibi-Témiscamingue administrative region. Figure 4-1 shows the location of the Project in the province of Quebec.

The Project is situated on NTS map sheet 32D/11. The approximate coordinates of its centre are 48°30'34"N, 79°12'34"W (UTM projection: 5374410N, 631517E, NAD83 Zone 17).

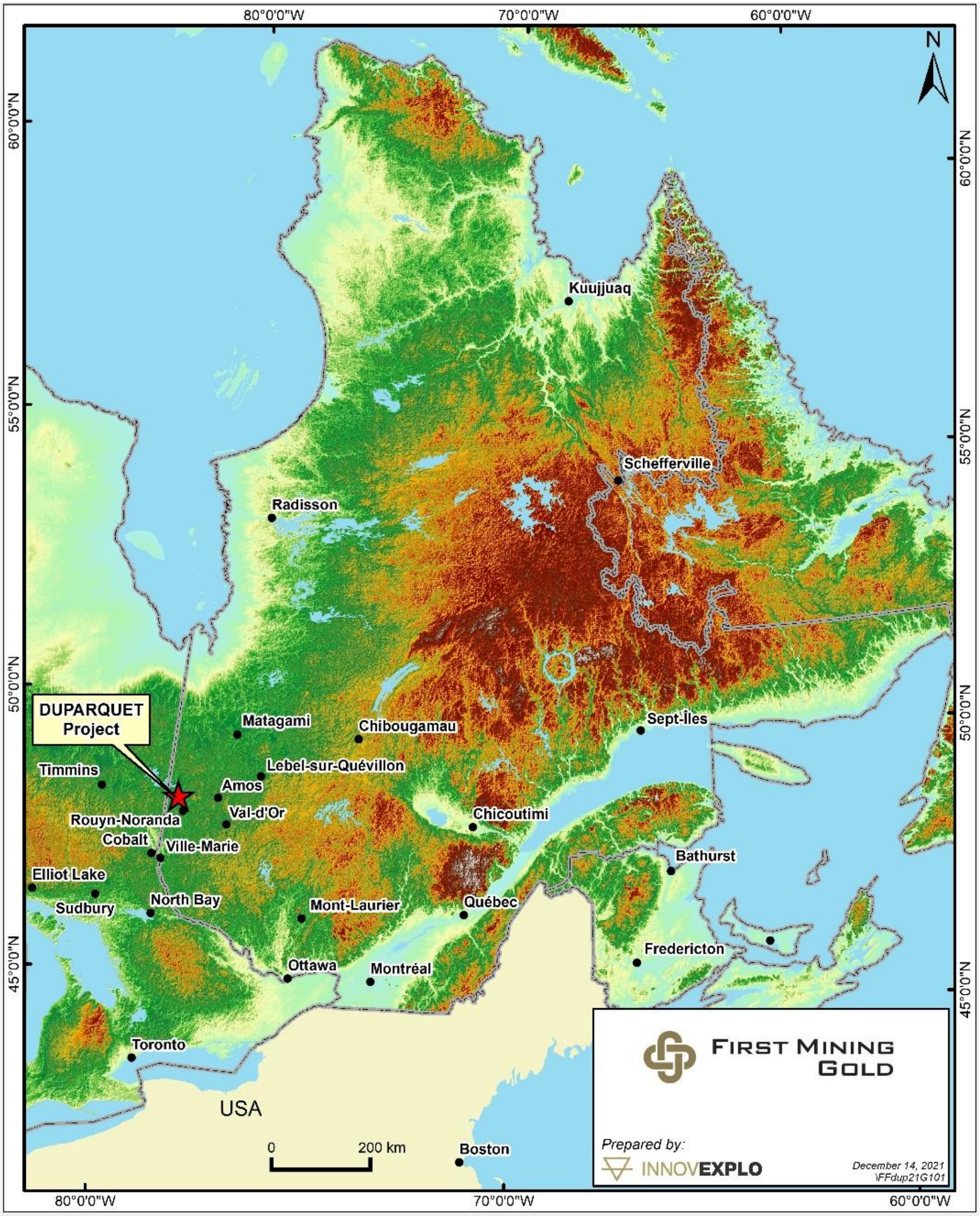


Figure 4-1 – Location of the Duparquet Project

As defined in this report, the Project includes the four amalgamated properties plus the area covered by tailings, which straddles the southwestern limit of Beattie (Figure 4-2).

Of the total surface area covered by the Project (1,079.2 ha), Beattie accounts for 383.6 ha, Donchester 322.6 ha, Central Duparquet 338.6 ha, and Dumico the remaining 34.4 ha.

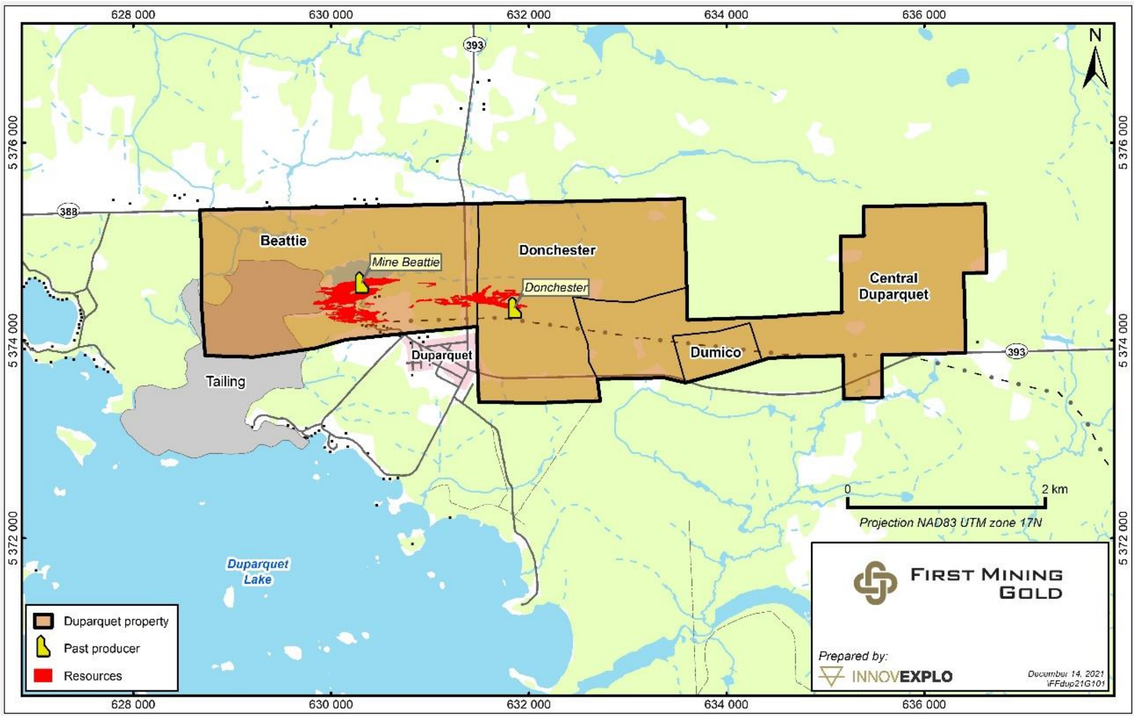


Figure 4-2 – Map of the Duparquet Project showing the subdivisions of the Beattie, Donchester, Central Duparquet and Dumico properties and the extent of the tailings

4.2 Mining Title Status

InnovExplo verified the status of all mining titles in GESTIM.

The Project comprises fifty (50) map-designated claims ("CDC") covering an area of 1,079.2 ha. It is comprised of 4 contiguous mining properties which include Beattie, Donchester, Dumico and Central Duparquet.

The mining concessions for the previously registered Beattie (CM292) and Donchester (CM442) properties were voluntarily allowed to lapse in April 2021 and August 2021 respectively and have since been converted to mining claims (CDC) by the owners. The change from Mining Concession to Mining Claims are the result of an amendment to the Mining Act, Bill 70, Chapter M-13-1 relating to non mining operational mining concessions.

Figure 4-3 presents the mineral title map, and Table 4-1 lists the mineral titles with ownership.

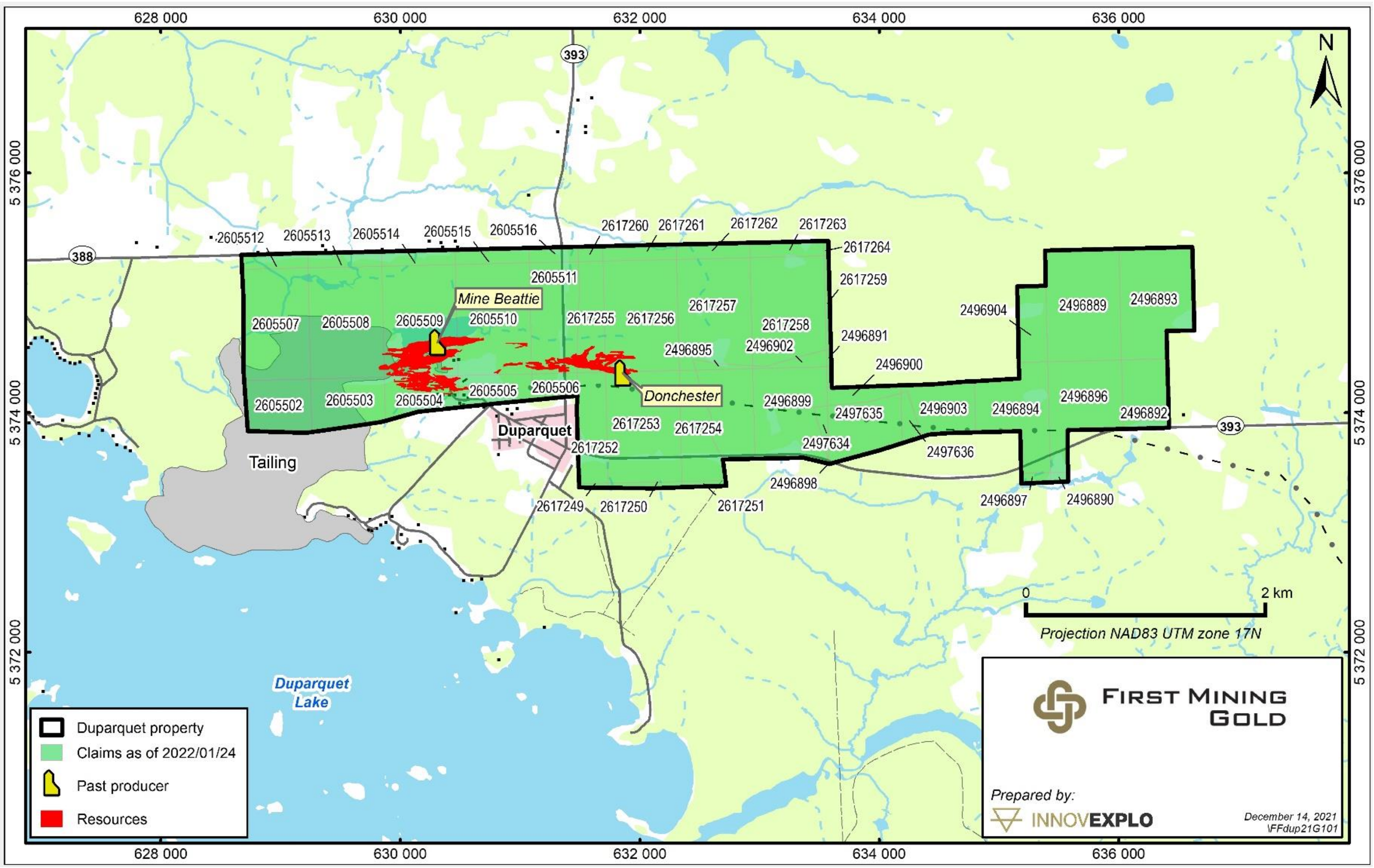


Figure 4-3 – Mining title and land-use map for the Duparquet Porperty

Table 4-1 – List of mining titles

No. Title	Area (Ha)	Status	Registration Date	Expiration Date	Owner	Comment
CDC2496889	57.03	Actif	2017-07-18	2024-07-24	Mines d'Or Duquesne Gold Mines Inc	
CDC2496890	1.15	Actif	2017-07-18	2024-07-24	Mines d'Or Duquesne Gold Mines Inc	
CDC2496891	0.58	Actif	2017-07-18	2024-07-24	Mines d'Or Duquesne Gold Mines Inc	
CDC2496892	22.14	Actif	2017-07-18	2024-07-24	Mines d'Or Duquesne Gold Mines Inc	
CDC2496893	51.66	Actif	2017-07-18	2024-07-24	Mines d'Or Duquesne Gold Mines Inc	
CDC2496894	38.66	Actif	2017-07-18	2024-07-24	Mines d'Or Duquesne Gold Mines Inc	
CDC2496895	2.4	Actif	2017-07-18	2024-07-24	Mines d'Or Duquesne Gold Mines Inc	
CDC2496896	40.69	Actif	2017-07-18	2024-07-24	Mines d'Or Duquesne Gold Mines Inc	
CDC2496897	2.03	Actif	2017-07-18	2024-07-24	Mines d'Or Duquesne Gold Mines Inc	
CDC2496898	0.01	Actif	2017-07-18	2024-07-24	Mines d'Or Duquesne Gold Mines Inc	
CDC2496899	44.28	Actif	2017-07-18	2024-07-24	Mines d'Or Duquesne Gold Mines Inc	
CDC2496900	8.04	Actif	2017-07-18	2024-07-24	Mines d'Or Duquesne Gold Mines Inc	
CDC2496901	28.46	Actif	2017-07-18	2024-07-24	Mines d'Or Duquesne Gold Mines Inc	
CDC2496902	4.3	Actif	2017-07-18	2024-07-24	Mines d'Or Duquesne Gold Mines Inc	
CDC2496903	21.93	Actif	2017-07-18	2024-07-24	Mines d'Or Duquesne Gold Mines Inc	
CDC2496904	15.23	Actif	2017-07-18	2024-07-24	Mines d'Or Duquesne Gold Mines Inc	
CDC2497634	2.65	Actif	2017-09-12	2024-05-14	173714 Canada Inc	Affected by the July 18, 2022 offer
CDC2497635	27.17	Actif	2017-09-12	2024-05-14	173714 Canada Inc	Affected by the July 18, 2022 offer
CDC2497636	4.59	Actif	2017-09-12	2024-05-14	173714 Canada Inc	Affected by the July 18, 2022 offer
CDC2605502	23.77	Actif	2021-04-08	2024-04-07	Beattie Gold Mines Ltd	Affected by the July 18, 2022 offer
CDC2605503	24.98	Actif	2021-04-08	2024-04-07	Beattie Gold Mines Ltd	Affected by the July 18, 2022 offer
CDC2605504	18.63	Actif	2021-04-08	2024-04-07	Beattie Gold Mines Ltd	Affected by the July 18, 2022 offer
CDC2605505	14.87	Actif	2021-04-08	2024-04-07	Beattie Gold Mines Ltd	Affected by the July 18, 2022 offer
CDC2605506	7.74	Actif	2021-04-08	2024-04-07	Beattie Gold Mines Ltd	Affected by the July 18, 2022 offer

No. Title	Area (Ha)	Status	Registration Date	Expiration Date	Owner	Comment
CDC2605507	51.11	Actif	2021-04-08	2024-04-07	Beattie Gold Mines Ltd	Affected by the July 18, 2022 offer
CDC2605508	57.03	Actif	2021-04-08	2024-04-07	Beattie Gold Mines Ltd	Affected by the July 18, 2022 offer
CDC2605509	57.03	Actif	2021-04-08	2024-04-07	Beattie Gold Mines Ltd	Affected by the July 18, 2022 offer
CDC2605510	57.03	Actif	2021-04-08	2024-04-07	Beattie Gold Mines Ltd	Affected by the July 18, 2022 offer
CDC2605511	36.87	Actif	2021-04-08	2024-04-07	Beattie Gold Mines Ltd	Affected by the July 18, 2022 offer
CDC2605512	6.75	Actif	2021-04-08	2024-04-07	Beattie Gold Mines Ltd	Affected by the July 18, 2022 offer
CDC2605513	7.55	Actif	2021-04-08	2024-04-07	Beattie Gold Mines Ltd	Affected by the July 18, 2022 offer
CDC2605514	7.56	Actif	2021-04-08	2024-04-07	Beattie Gold Mines Ltd	Affected by the July 18, 2022 offer
CDC2605515	7.58	Actif	2021-04-08	2024-04-07	Beattie Gold Mines Ltd	Affected by the July 18, 2022 offer
CDC2605516	5.06	Actif	2021-04-08	2024-04-07	Beattie Gold Mines Ltd	Affected by the July 18, 2022 offer
CDC2617249	0.83	Actif	2021-08-18	2024-08-17	173714 Canada Inc	Affected by the July 18, 2022 offer
CDC2617250	2.91	Actif	2021-08-18	2024-08-17	173714 Canada Inc	Affected by the July 18, 2022 offer
CDC2617251	1.73	Actif	2021-08-18	2024-08-17	173714 Canada Inc	Affected by the July 18, 2022 offer
CDC2617252	22.55	Actif	2021-08-18	2024-08-17	173714 Canada Inc	Affected by the July 18, 2022 offer
CDC2617253	57.04	Actif	2021-08-18	2024-08-17	173714 Canada Inc	Affected by the July 18, 2022 offer
CDC2617254	24.03	Actif	2021-08-18	2024-08-17	173714 Canada Inc	Affected by the July 18, 2022 offer
CDC2617255	20.16	Actif	2021-08-18	2024-08-17	173714 Canada Inc	Affected by the July 18, 2022 offer
CDC2617256	57.03	Actif	2021-08-18	2024-08-17	173714 Canada Inc	Affected by the July 18, 2022 offer
CDC2617257	54.63	Actif	2021-08-18	2024-08-17	173714 Canada Inc	Affected by the July 18, 2022 offer
CDC2617258	52.73	Actif	2021-08-18	2024-08-17	173714 Canada Inc	Affected by the July 18, 2022 offer
CDC2617259	3.1	Actif	2021-08-18	2024-08-17	173714 Canada Inc	Affected by the July 18, 2022 offer
CDC2617260	2.54	Actif	2021-08-18	2024-08-17	173714 Canada Inc	Affected by the July 18, 2022 offer
CDC2617261	7.62	Actif	2021-08-18	2024-08-17	173714 Canada Inc	Affected by the July 18, 2022 offer
CDC2617262	7.62	Actif	2021-08-18	2024-08-17	173714 Canada Inc	Affected by the July 18, 2022 offer
CDC2617263	7.62	Actif	2021-08-18	2024-08-17	173714 Canada Inc	Affected by the July 18, 2022 offer

No. Title	Area (Ha)	Status	Registration Date	Expiration Date	Owner	Comment
CDC2617264	0.48	Actif	2021-08-18	2024-08-17	173714 Canada Inc	Affected by the July 18, 2022 offer

Beattie, Donchester, Dumico Properties and Surface Rights

The Beattie, Donchester and Dumico properties were held by three (3) companies.

Beattie Gold Mines Ltd ("Beattie Gold Mines"), having its registered office at 2147 Portage Avenue, Winnipeg, Manitoba, was the legal and beneficial owner of a 100% undivided interest in the mineral mining rights of the previously registered Beattie mining concession (historical MC#292 of the Beattie Project).

2588111 Manitoba Ltd ("2588111"), having its registered office at 2147 Portage Avenue, Winnipeg, Manitoba, owned, through its wholly owned subsidiary, 173714 Canada Inc ("173714"), the mineral mining rights of the previously registered Donchester mining concession (historical MC#384 of the Donchester Project), the mineral mining and surface rights of the previously registered Hunter mining concession (historical MC#442 of the Hunter Project) and mining rights on the Dumico Claims (historical claims C003231 and C003232).

2699681 Canada Ltd ("2699681"), having its registered office at 2147 Portage Avenue, Winnipeg, Manitoba, owned, through its wholly owned subsidiary, Eldorado Gold Mines Inc. ("Eldorado"), the surface rights to the previously registered Beattie mining concession #292, including the concentrate roaster located thereon, but excepting land that has been sold for the golf course, the houses near the golf course with their accompanying land, and the northeast part of the previously registered MC#292. Eldorado also owns part of the surface rights to the previously registered Donchester mining concession #384, excepting land that has been deeded to the church for the cemetery, and the northwest part of the previously registered MC#384. Eldorado owns the surface rights to the Dumico Project. Eldorado also owns the mine tailings that originated from the original Beattie, Donchester, Duquesne and Hunter mines.

On May 1, 2008, Clifton Star Resources Inc. ("Clifton Star") signed mineral Project option agreements with the three companies presented above, with similar terms. The initial agreements have been amended three times (July 22, 2008, November 24, 2008, and April 8, 2009). On October 26, 2009, Clifton Star signed a Letter of Intent with the optionors. Clifton Star then renegotiated with the optionors to terminate the aforementioned mineral Project option agreements and enter new agreements, which resulted in payments of $600,000 to Beattie Gold Mines, $300,000 to 2699681, and $600,000 to 2588111 under the old option agreements (Clifton Star, MD&A, May 29, 2012). On September 12, 2012, Clifton Star announced in a news release that new terms had been negotiated with the optionors. On December 1, 2014 Clifton Star announced that it would not make the $10 million option payment on December 1st, 2014, that would have enabled it to continue the Duparquet Option Agreements. This $10 million payment was part of a total of $52.2 million in option payments required to be made by Clifton Star by 2017 in order to allow Clifton Star to acquire the 90% interest in the Duparquet Project that it does not already own. All payments were necessary to acquire the additional 90% interest. Clifton Star retained a 10% interest in the companies that own the Duparquet Project.

On April 8, 2016, First Mining announced the successful completion of the business combination, pursuant to which First Mining had acquired all of the issued and outstanding shares of Clifton Star by way of a court approved plan of arrangement. Clifton Star's shares were de-listed from the TSX Venture Exchange and it ceased being a reporting issuer under applicable Canadian securities laws.

On February 8, 2022 First Mining acquired, from two individuals, an aggregate of 286,904 common shares of Beattie Gold Mines. The per Share consideration paid under the Transaction was for a total consideration of C$1,272,824 in cash and 7,636,944 First Mining common shares. Together with the 187,839 common shares of Beattie Gold Mines already owned by Clifton Star, a wholly-owned subsidiary of First Mining, the Company now owns 474,743 shares of Beattie Gold Mines, increasing its ownership from 10% to 25.3% of the issued and outstanding common shares of Beattie Gold Mines.

On September 15, 2022, First Mining acquired all the issued and outstanding shares of Beattie Gold Mines, 2588111 and 2699681. This purchase represents a majority interest in the companies that make up Beattie, Donchester, Dumico properties, as well as Eldorado which owns the surface rights and tailings.

The total acquisition cost is approximately C$24 million to acquire Beattie Gold Mines and 2588111, which own the mineral rights to Beattie, Donchester and Dumico includes C$8,727,177 in cash and 69,127,820 common shares of First Mining.

- For Beattie Gold Mines: Acquisition cost of C$16.9 million consisting of:
  - 49,127,820 Shares of First Mining valued at C$10.8 million based on the 20-Day Volume Weighted Average Price (“VWAP”);
  - C$ 6 227 176 in cash.
- For 2588111: Acquisition cost of C$6.9 million consisting of:
  - 20 million Shares of First Mining valued at C$4.4 million based on the 20-Day VWAP;
  - C$2,500,000 in cash.

Central Duparquet Property

The Central Duparquet Property at the time of purchase was defined as a group of eighteen (18) contiguous claims totalling 293 ha, registered to Gilles Fiset and Lizette Grenier. On December 15, 2008, Clifton Star signed an option agreement whereby it could acquire a 100% interest in the Central Duparquet Project. To earn its 100% interest, Clifton Star paid $400,000 on January 13, 2009. On May 2nd, 2013, the Company paid $125,000 in cash to extend the five-year period to six years and six months. During fiscal 2015, the Company relinquished its interest in the Project.

In February 2017, First Mining had acquired eighteen mining claims located in the Township of Duparquet, Québec from a private individual in exchange for $250,000 and 2,500,000 First Mining shares.

On February 26, 2010, Clifton Star entered into an agreement to acquire the 2% NSR from the Gesmalar, the original owner of the Central Duparquet Project. As consideration for the acquisition of the NSR, Clifton Star paid $155,000 and issued 10,000 common shares, valued at $57,400, to Gesmalar.

The Central Duparquet Project has since been converted from claims (“CL”) to map staked mining claims (“CDC”) and now includes 16 map designated claims covering an area of 338.6 ha which are registered under the name of Mines d’Or Duquesne Gold Mines Inc., 100% owned by First Mining.

4.3 Socio-Environmental Responsibilities

The Duparquet Project is located within the urban perimeter of the municipality of Duparquet (Figure 4-4) but is not affected by restrictions on exploration and mining activities under Quebec's *Mining Act* or *Act respecting Land Use Planning and Development*.

According to the *Mining Act*, any mineral substance forming part of the domain of the State and found in an urban perimeter shown on maps kept at the registrar's office, except mineral substances found in a territory subject to a mining right obtained before December 10, 2013, is withdrawn from prospecting, mining exploration and mining operations as of that date, until the territories provided for in section 304.1.1 of the *Mining Act* and section 6 (par. 7) of the *Act respecting Land Use Planning and Development* (mining-incompatible territories) are determined by decision of the RCM. A mining-incompatible territory is a territory in which the viability of activities would be compromised by the impacts of mining. As the RCM did not define CM292 as a mining-incompatible territory before December 14, 2016, the date on which the Act respecting Land Use Planning and Development came into force, the issuer is allowed to carry out exploration and mining activities on the Beattie mining concession or its derivative.

The area covered by tailings (the original tailings and waste rock management area plus subsequent tailings spread) partially overlies Crown land (pers. comm. with Bryan Goulet of the Abitibi-Ouest MRC on February 26, 2013). The Ministry of Energy and Natural Resources of Quebec (the "MERN") possesses the clearing rights. The issuer must purchase the surface rights from the MERN if they want to recover the portion of the tailings outside the Project. ***There is a risk that the MERN will not allow the issuer to recover the mineral resource contained in the tailings that have spread beyond the Project. However, this represents a negligible portion of the Project's total mineral resources. It would have no material impact on the MRE and would not affect the potential viability of the Project***.

InnovExplo is not aware of any environmental liabilities, permitting issues or municipal social issues concerning the Project. All exploration activities conducted on the Project comply with the relevant environmental permitting requirements.

4.4 Permits

The Project is located in Abitibiwinni First Nation territory (Figure 4-4) Claim holders within this territory are bound by the *Agreement on Consultation and Accommodation* between the Abitibiwinni First Nation Council and the Government of Quebec.

In general, the recommended work program in this report requires minimal permitting. However, if drilling is to be carried out, the issuer must obtain specific permits and authorizations from the relevant government agencies, including a timber permit (*Autorisation de coupe de bois sur un territoire du domaine de l'État où s'exerce un droit minier*) from the MERN.

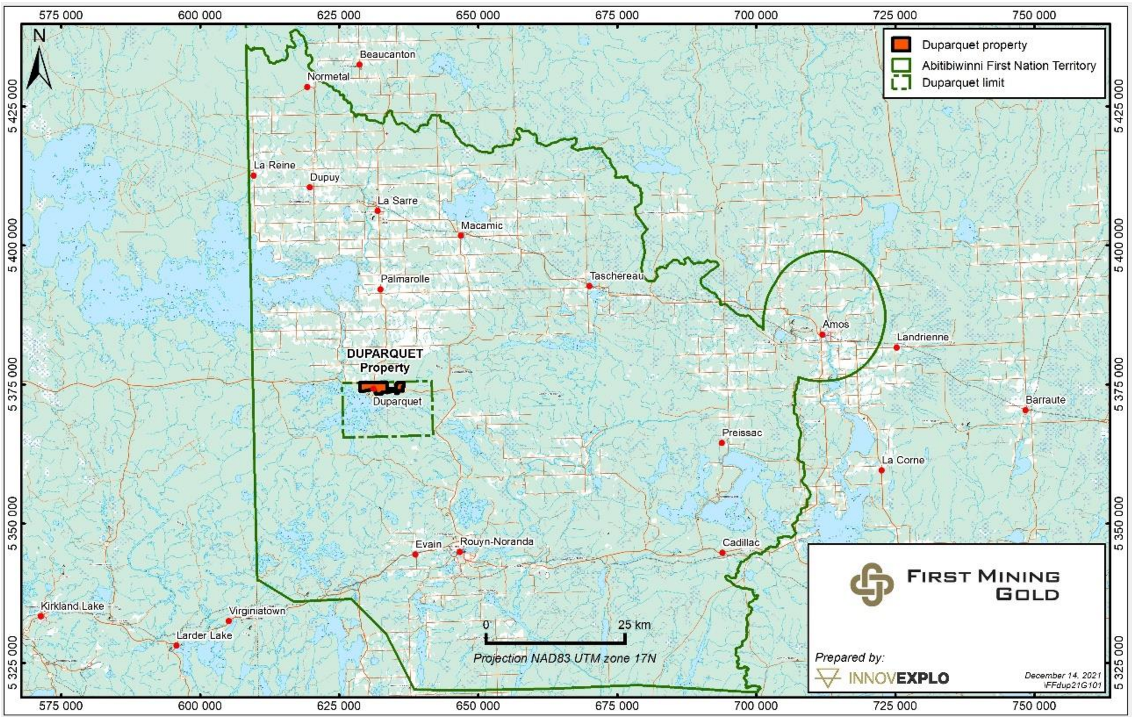


Figure 4-4 – Duparquet Project location in the Abitibiwinni First Nation territory

4.5 Social Licence Considerations

Social acceptance will be necessary for the project's success given its location in the Abitibiwinni First Nation territory and inside the municipal limits of Duparquet. In addition, some parts of the Project are private lands with houses and a golf course and clubhouse.

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

5.1 Accessibility

The Project is located just north of the town of Duparquet, which can be reached by paved, two-lane, all-season provincial highways from Rouyn-Noranda (53 km to the south; Route 393 and Route 101) or La Sarre (33 km to the north; Route 393) (Figure 5-1). The highways pass through parts of the Project and several gravel roads also lead onto it.

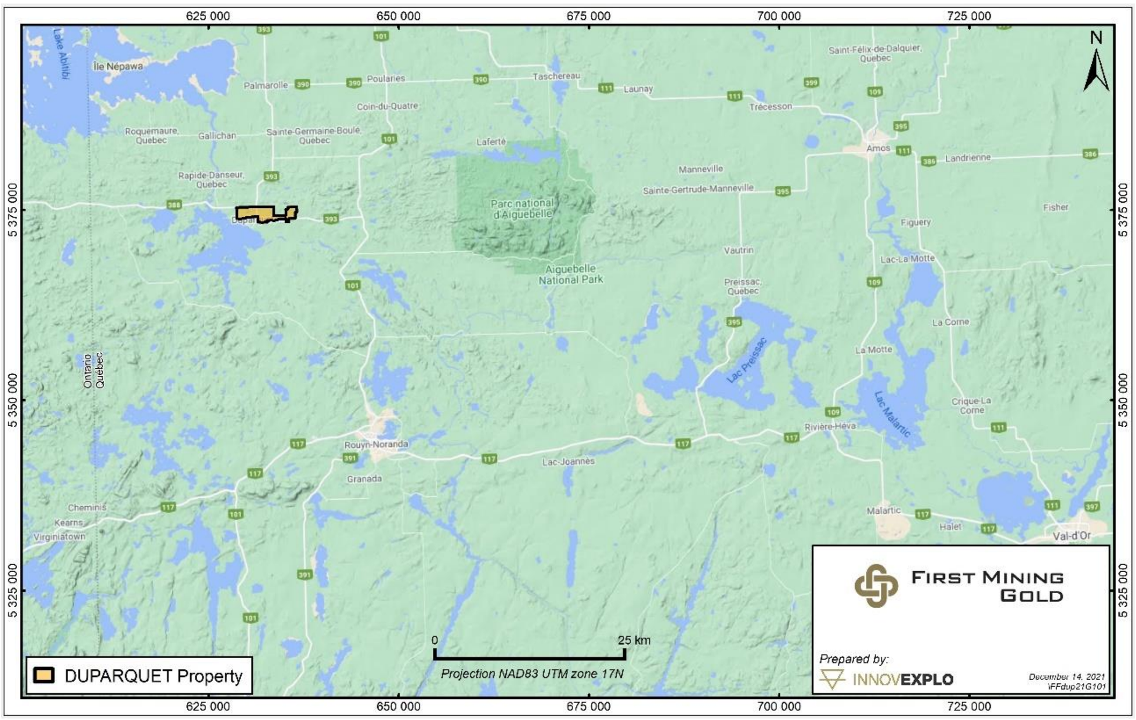


Figure 5-1 – Access to the Duparquet Project

5.2 Climate

The climate is continental, with cold, dry winters and warm summers. Winter temperatures average -17°C with lows down to -40°C in January, whereas summer temperatures average 17°C with highs up to 35°C in July. Annual precipitation is around 900 mm. Snow falls from mid-November to mid-April.

Mining and drilling operations can be conducted year-round. Surface exploration work, such as mapping and channel sampling, can generally only be carried out from mid-April to mid-November.

5.3 Local Resources

Rouyn-Noranda is the nearest major city with a specialized workforce and facilities for obtaining mining/exploration repairs and supplies. The town of Duparquet currently has a population of about 711 residents, many of whom are family members and descendants of workers at the past-producing Beattie and Donchester mines. Skilled workers and housing are available in the municipality. Water is available from the town's water supply or bodies of water on the Project.

Electric power is available at the Beattie mine site. In 2013, Hydro-Québec completed an exploratory study to verify its capacity to supply electric power to the Project (Williamson et al., 2013a). It is possible to feed the Project from the Renaud substation, but a new 120 KV line, 14.5 km long, should be built to connect the Project. Hydro-Québec would require 36 months to complete the power supply.

5.4 Infrastructure

Most of the mine buildings have been demolished. The shafts and vent raises have been capped and allowed to flood naturally. The existing connection at surface to underground workings has also flooded. The only remaining buildings on the Beattie mine site are the roaster, smokestack and water tower.

Parts of the roaster building are currently used as an office, core shack and pulp and reject storage by the issuer. The existing infrastructure is in a poor state of repair and its demolition and remediation would be expected to form part of the development. The yard holds numerous core racks and now serves as a storage area for cores, pulps and rejects. A locked gate restricts access to these facilities.

5.5 Physiography

The Project lies in moderately rolling terrain just south of a flat belt of glacial till deposits and farmland. Outcrop density varies from 20% to 50%, but up to 80% in specific areas. The area is covered with an immature to semi-mature forest of poplar with some birch, balsam, spruce and jack pine interspersed on well-drained higher ground. The Project is also characterized by dense and thick undergrowth consisting mainly of tag alders, scrub maple and willow. The overburden consists of sandy soil or till, with occasional gravel fractions. Some granitic boulders, up to 3 m in diameter, are scattered across the surface of the land.

The mean elevation of the Project is approximately 300 m above sea level (masl).

As described in Fillion (2009), the mine tailings are contained within a well-defined naturally occurring topographic basin that prevents them from spreading, except along the shore of Duparquet Lake, where they have spread over a distance of 300 m into the water on a very gentle slope. The tailings overlie lacustrine clay. A cover of organic matter referred as "gumbo clay", sometimes containing fire residue, is often present between the tailings and the lacustrine clay. Many intermittent creeks appear and then dry out over the course of the year in the general area of the tailings. One main trail crosses the entire area, reaching the lake, and can be used year-round. Vegetation grows slowly on the tailings surface and consists of several tree species, including birch, poplar and spruce. A marsh is present in the northwest part of the Project. A permanent waterway isolates the western part of the tailings area.

6. HISTORY

The following chronological summary of historical work on the former properties forming the current Project was compiled from Bevan (2011) and Dupéré et al. (2011) and reviewed by InnovExplo.

6.1 Historical work

Table 6-1– Summary of historical work carried out on the Duparquet Project

Year	Company	Work	Results
1910	John Beattie	First gold discovery in Duparquet Township	
1923		Staking of the Beattie Project	
1924	Victoria Syndicate	Trenching on Beattie Project	No significant results
1925-1927	Consolidated Mining and Smelting Company of Canada Ltd	Beattie Project: Optioning of claims Exploration and prospecting of claims Trenching Drilling program	No significant results
1930			Dropped option
1930	John Beattie Ventures Limited and Nipissing Mining Company Ltd	Beattie Project: Exploration Drilling program Sinking of shaft to 67 m	Discovery of "North" mineralized zone Discovery of "A" mineralized zone
1931-1945	Dumico Gold Corp.	Central Duparquet: Drilling program (52 DDHs) to define the mineralized zone	Sinking of exploration shaft Driving of 5 levels
1932	Beattie Gold Mines Ltd	Development starts at the Beattie mine: 6-compartment shaft sunk to 442 m Development of 9 levels at 46-m intervals	
1933		Production starts at the Beattie mine: Flotation process set up	Concentrate shipped to smelter in Tacoma, Washington
1934-1937		Cyanidation plant installed (1935) Roaster added to improve recoveries (1937)	Initial production at 800 short tons per day Maximum production of 1,900 short tons per day
1937		Compartment winze sunk from level 5 (244 m) down	

Year	Company	Work	Results
		to level 13 (610 m)	
1938-1940	Beattie Gold Mines (Québec) Ltd	Drift driven from the third level of the Beattie mine across the boundary with the Donchester Project	
1941		Drilling program on Donchester Project Donchester mine purchased Development of drift from the Beattie shaft (6th level) to Dumico shaft on Central Duparquet Project	9 DDHs Intersections ranging from 0.15 to 0.40 oz/t Au Vein identified with 792 m strike length at 0.28 oz/t Au over an average width of 2 m
1943-1944		Shaft sunk on Donchester Project to 6th level	
1943		Cave-in at Beattie mine: failed main crown pillars	Rehabilitation work Production drops
1943-1950		Production commenced on Donchester section Beattie mine stays open during post-war years	Production loss
1945		Donchester shaft deepened to 9th (457 m) level Development of 7 levels	At least 4 levels driven across to the North Zone deposit
1946-1956	Consolidated Beattie Mines Ltd	Company re-organization; name change (1946) Production ceased (1956)	Total production of the Beattie mine: 10,614,421 t with an average grade of 0.12 oz/t Au. From that total, 1,350,000 t at 0.14 oz/t Au were from the Donchester section
1956-1987		Beattie Project: Exploration program including line cutting, EM survey and drilling (259 m) to test an EM anomaly Project dormant	EM anomaly found to be caused by graphite Old mine records destroyed by fire and affected by water damage. The latest complete records of reserves and production (stope outlines) are those on a longitudinal section dated January 1, 1954.
1981	SOQUEM Inc.	Central Duparquet Project: Drilling program (13 DDHs)	
1987	SOQUEM Inc. Cambior Inc.	Central Duparquet Project: Mapping	Re-definition of Project reserves

Year	Company	Work	Results
		Lithogeochemical survey Geophysical survey 18 surface trenches 3 drilling programs	Various feasibility studies: open pit methods suggested
1988	Beattie Gold Mines Ltd	Drilling program (12 DDHs totalling 1,939 m) targeting the "A", South and North Zones	
1989-1990	Forbex Mining Resources Inc.	Central Duparquet Project: Reserves update Drilling program (11 DDHs) west of known mineralized zones	
1989-1994	Beattie Gold Mines Ltd	Drilling programs on the South Zone (23 DDHs totalling 2,077 m)	
1994-1995	Fieldex Inc.	Drilling program (6 DDHs) on the eastern portion of Central Duparquet Project	
1995	Beattie Gold Mines Ltd	Drilling program (3 DDHs totalling 284 m) on the South Zone and at depth	
1996		Drilling program (7 DDHs totalling 626 m) on the North Zone–East Extension and South Zone D-vein	
1997-2001		Drilling program (9 DDHs with one extension totalling 1,815 m) on the North Zone of the Beattie and Donchester properties	
2002-2003	Beattie Gold Mines Ltd	Drilling program (6 DDHs totalling 839 m) on the South Zone veins A, C, D and E	
2004	9085-3615 Québec Inc.	Pilot project for mining the Central Duparquet Project	No significant results
	Beattie Gold Mines Ltd	Drilling program (extension of 2 DDHs drilled in 2002 and 2003; total of 246 m)	
2005	Golder Associates Beattie Gold Mines Ltd	Drilling program (1 DDH totalling 313 m) on the South Zone	
2006-2007	Beattie Gold Mines Ltd	Drilling program (6 DDHs totalling of 578 m) on the South Zone	
2008-2009	Clifton Star Resources Inc.	Drilling programs:	

Year	Company	Work	Results
		209 DDHs totalling 58,053 m on the Beattie Project 99 DDHs totalling 37,566 m on the Donchester Project 19 DDHs totalling 4,818 m on the Dumico Project	
2010	Osisko Mining Corp. and Clifton Star Resources Inc.	Drilling program (314 DDHs totalling 102,529 m) on the Donchester and Beattie properties Channel sampling (220 channels totalling 460 m)	
2010	SGS Mineral Services (Lakefield, Ontario) Osisko Mining Corp. and Clifton Star Resources Inc.	Comminution testwork and preliminary cyanidation and flotation tests	Abrasion index moderate to high Maximum gold recovery of 41.6%
2011	Geophysics GPR International Inc. Osisko Mining Corp. and Clifton Star Resources Inc.	Helicopter-borne Mag and TDEM geophysical survey	Entire Project covered by the survey
2011	Clifton Star Resources Inc.	Drilling program (46 DDHs totalling 17,565 m) on the Beattie, Donchester and Central Duparquet properties	
2012	Clifton Star Resources Inc.	Stripping of 19 outcrops on Beattie Project, 9 on Donchester, 1 on Central Duparquet Channel sampling (719 samples)	
2012	Clifton Star Resources Inc.	Re-sampling of 50 DDHs for untested shoulders adjacent to mineralized zones (4,025 new samples)	
2012	Clifton Star Resources Inc.	Drilling program (53 new DDHs totalling 15,901 m and 18 extensions of older DDHs) on Beattie, Donchester and Central Duparquet properties	
2012	Tenova Mining & Minerals – Bateman Engineering Pty Ltd Clifton Star Resources Inc.	Preliminary CAPEX and OPEX costs for the construction and operation of a mineral processing plant to produce mineralized material from the Project to produce gold doré	Pre-production capital and sustaining costs for the Project estimated at \$370 M and \$144 M, respectively, excluding \$22.6 M for closure costs. Average operating cash cost estimated at

Year	Company	Work	Results
			US$726/oz Au.
2012	SGS Canada Inc. (Lakefield, Ontario) Clifton Star Resources Inc.	Metallurgical study: flotation, pressure oxidation and cyanidation testwork to investigate the recovery of gold from ore and tailings samples.	Bond work index of ore samples from 17.2 kWh to 20.2 kWh/t classify them as hard to very hard Preliminary gravity separation tests yielded gold recoveries from 3.7 to 14.9%, averaging 8.6% Gold recovery for bulk sulphide concentrate by flotation greater than 90% Overall gold recoveries from 83.5% to 93.3% for tailings samples
2012	Clifton Star Resources Inc.	15 of the 19 DDHs on the Dumico Project were entirely resampled and/or downhole surveys performed	
2013	SGS Canada Inc. (Lakefield, Ontario) Clifton Star Resources Inc.	Metallurgical and environmental pilot tests on a 12 t composite drill core bulk sample from the mineralized zones A pilot plant went into operation to generate a bulk sulphide flotation concentrate analyzing 15-18% S for a subsequent pressure oxidation pilot plant to recover gold	With one cleaning stage the gold recovery was 91.7% in a concentrate that assayed 26.8 g/t Au and 16.1% S Overall gold recovery was 95.4% with one cleaning stage and 91.9% with two cleaning stages
2013	Clifton Star Resources Inc.	41 outcrops stripped Channel sampling (1,001 samples) Re-sampling of 8 of the company's DDHs for untested shoulders adjacent to mineralized zones (397 new samples) Drilling program (92 DDHs totalling 16,773.5 m) on Beattie, Donchester and Central Duparquet properties	

1910 Gold is first discovered in Duparquet Township by John Beattie.

1923 The first claims for the Beattie property, included in Mining Concession 292, are staked by John Beattie, for whom the property is named.

1924 The Victoria Syndicate options the Beattie property and carries out extensive trenching, but results are not encouraging.

1925-1927 Prospecting of the Beattie property claims continues. *Consolidated Mining and Smelting Company of Canada* Ltd options the claims. The company continues exploration by trenching and carries out some drilling.

1930 Consolidated Mining and Smelting *Company of Canada* Ltd drops the option. In the same year, John Beattie discovers the Main (or North) deposit and options the property to Ventures Limited and Nipissing Mining Company Ltd. These two companies advance capital to develop the Beattie mine. The North deposit is drilled and a two-compartment shaft is sunk to 67 m. During the sinking of the shaft, another deposit is encountered, called the "A" mineralized material zone.

1931-1945 Exploration and development is carried out on the Central Duparquet property by Dumico Gold Corp., then Central Duparquet Mines Ltd. Contemporaneous with the start of production at the Beattie mine, further west, Dumico drills 52 surface holes to define the mineralized zone. This work justifies the sinking of an exploration shaft and the driving of five (5) levels.

1932 The operator of the Beattie mine, Beattie Gold Mines Ltd, is formed. A six-compartment (6) shaft is sunk to a depth of 442 m and nine (9) levels are established at 46 m intervals, with the first level at 61 m below the shaft collar.

1933 A 2,000 tpd flotation process plant is erected and production commences, with concentrates being shipped to Asarco's smelter in Tacoma, Washington.

1934-1937 A cyanidation plant is installed in 1935 and, due to the sulphide content in the mineralized material, a roaster is added in 1937 to improve recoveries. Initially, the production rate is 800 tpd, gradually building up to 1,500 tpd in 1935 to a maximum of 1,900 tpd.

1937 A three-compartment (3) winze or internal shaft is sunk from the 5th (244 m) level some 274 m east of the main shaft down to the 9th (427m) level. This winze is later deepened to 625 m with the 13th (610m) level established.

1939 Beattie Gold Mines Ltd is re-organized, becoming Beattie Gold Mines (Québec) Ltd.

1938-1940 Mine development is carried out in the Beattie mine toward the Donchester property. The Donchester property is immediately east of the Beattie property. A drift from the third level within the Beattie mine is driven across the boundary with the Donchester property for exploration purposes.

1933-1940 From the start of production in 1933 until the end of 1940, the mill treated 3,921,281 t of ore and recovered 471,085 oz of gold and 73,214 oz of silver, an average of 0.120 oz/t Au and 0.019 oz/t Ag (Dresser and Denis, 1949). This tonnage came from the North and A zones.

1941 Drilling from underground stations at 152-metre intervals outlines several ore-shoots 91 m below surface within the Donchester boundary. Nine (9) holes intersect values varying from 0.15 to 0.40 oz/t Au over widths ranging from 0.3 to 4 m. Beattie Gold Mines acquires the Donchester mine, for stock considerations. On the strength of these results, a drift on the 6th level (330 m below surface) is driven from the Beattie shaft over to the Dumico shaft of the Central Duparquet property, cutting across Donchester ground. Cross-drilling from this drift outlines a vein 792 m in strike length and grading 0.28 oz/t Au over a 2 m width on average. Peak production at the Beattie mine is reached in 1941 and 1942 at 1,900 tpd.

1943-1944 A shaft is sunk on Donchester ground to the 6th level and connected to the previously driven drift.

1943 A cave-in at the Beattie mine caused by failure of the main crown pillars results in an inrush of about one million cubic yards of clay, sand and broken rock into the mine workings. Rehabilitation work starts immediately and continues until 1950.

1943-1950 During this period, mining exploration in the original Beattie mine suffers and operations are conducted at a loss. Much of the production slack is taken up by tonnage from the Donchester section which is brought on-stream sooner because of the cave-in at Beattie. Production losses are accentuated during the war years by the shortage of labour and supplies, and after 1946, by rising costs and a fixed price for gold. Only with a government cost aid program (the *Emergency Gold Mining Act*, E.G.M.A.) are gold mines such as Beattie mine able to stay open in the post-war years.

1945 The Donchester shaft is deepened to the 9th level (457 m vertical). Development is carried out above and below the 6th level resulting in seven (7) new levels from the 2nd to the 9th on the “South Zone” deposit. At least four levels are driven across to the “North Zone” deposit: the 4th, 5th, 6th and 8th levels. The 3rd level is also driven across from Beattie to inter-connect with the North Zone.

1946-1956 The company again re-organizes and becomes Consolidated Beattie Mines Ltd. Operations continue on the Beattie property until 1956 when, after 23 years of almost continuous production, the mine closes. During its lifetime, the Beattie mill treated 9,645,000 t with an average grade of 4.01 g/t Au and 0.99 g/t Ag from the North zone of the Beattie mine and Donchester mine (Lavergne, 1985).

1956-1987 Except for a small surface exploration program in 1966, the Beattie property remains dormant from 1956 to 1987. The 1966 program consists of line cutting, an electro-magnetic survey and two (2) holes totalling 259 m, drilled to test an EM anomaly which is found to be caused by graphite. Unfortunately, some of the old mine records are destroyed by fire and affected by water damage. The latest complete records of reserves and production (stope outlines) are now those on a January 1, 1954, longitudinal section.

1981 The first period of activity on the Central Duparquet property takes place with SOQUEM Inc ("SOQUEM"). SOQUEM performed a drilling program totalling thirteen (13) holes.

1987 The second period of activity on the Central Duparquet property takes place, first with SOQUEM followed by Cambior Inc, both companies mainly concentrating their efforts on the western part of the property. Mapping, lithogeochemical and geophysical surveys are carried out as well as eighteen (18) surface trenches and three (3) drilling programs. The aim of this work is to define the mineralized zone more accurately than Dumico Gold Corp. following their discovery of the zone. The first two (2) drilling programs carried out by SOQUEM in 1981 lead to the re-definition of property reserves (see below). The 3rd program is a drilling program conducted by Cambior in an attempt to define a near-surface mineralized zone. Various feasibility studies suggest that the deposit could be mined by open pit methods.

1988 A drilling program is carried out by Beattie Gold Mines, comprising twelve (12) holes for a total of 1,939.4 m. The holes target the "A", South and North Zones.

1989-1990 Forbex Mining Resources Inc. re-vamps the reserves from the Central Duparquet property and undertakes an 11-hole drilling program in the eastern part of the property, off the known mineralized zones to the west.

1989 Beattie Gold Mines drills ten (10) holes for a total of 401.7 m. These holes are drilled under a stripped area of the South Zone and to the west of the stripping.

1990 A drilling program is carried out by Beattie Gold Mines, comprising three (3) holes for a total of 713.8 m. The holes target the South Zone.

1991 A drilling program is carried out by Beattie Gold Mines, comprising two (2) holes for a total of 200 m. The holes target the South Zone.

1992 A drilling program is carried out by Beattie Gold Mines, comprising one (1) hole for a total of 185.6 m. This hole targets the South Zone.

1993 A drilling program is carried out by Beattie Gold Mines, comprising four (4) holes for a total of 277.4 m. The holes target the South Zone.

1994 A drilling program is carried out by Beattie Gold Mines, comprising three (3) holes for a total of 298.7 m. The holes target the South Zone.

1994-1995 Fieldex Inc drills six (6) holes in the eastern portion of the Central Duparquet property.

1995 A drilling program is carried out by Beattie Gold Mines, comprising three (3) holes for a total of 284 m. The holes target the South Zone extension to the east and at depth below earlier holes.

1996 A drilling program is carried out by Beattie Gold Mines, comprising seven (7) holes for a total of 625.8 m. The holes target the North Zone–East Extension with three (3) holes on the Beattie property and four (4) holes on the Donchester property. Two (2) holes (total of 62 m) are drilled into the South Zone, targeting the D-Vein on the Beattie property to the east of previous drill holes.

1997 A drilling program is carried out by Beattie Gold Mines, comprising three (3) holes for a total of 477 m. The holes target the North Zone–East Extension with one (1) hole on the Beattie property and two (2) holes on the Donchester property.

1998 A drilling program is carried out by Beattie Gold Mines, comprising three (3) holes for a total of 537 m. The holes target the North Zone–East Extension with one (1) hole on the Beattie property and two (2) holes on the Donchester property.

1999 A drilling program is carried out by Beattie Gold Mines, comprising two (2) holes for a total of 294 m. The holes target the North Zone–East Extension.

2000 A drilling program is carried out by Beattie Gold Mines, comprising one (1) hole for a total of 304 m. The hole targets the North Zone–East Extension.

2001 A drilling program is carried out by Beattie Gold Mines, this was an extension of the hole drilled in 2000 for a total of 203 m.

2002 A drilling program is carried out by Beattie Gold Mines, comprising three (3) holes for a total of 325 m. The holes target the South Zone and intersect the A to D veins.

2003 A drilling program is carried out by Beattie Gold Mines, comprising three (3) holes for a total of 516 m. The holes target the South Zone and intersect the C to E veins.

2004 The owner of the Central Duparquet property, 9085-3615 Québec Inc, embarks on a pilot project with the objective of mining the property. No record has been found documenting this work. A drilling program is carried out by Beattie Gold Mines representing the extension of two (2) holes drilled in 2002 and 2003 for a total of 246 m.

2005 A drilling program is carried out by Beattie Gold Mines, comprising one (1) hole for a total of 313 m. The hole targets the east extension of the South Zone.

2005-2007 Drilling continues on the Central Duparquet property, comprising seven (7) holes for a total of 891 m.

2006 A drilling program is carried out by Beattie Gold Mines, comprising one (1) hole for a total of 294 m into the South Zone.

2007 A drilling program is carried out by Beattie Gold Mines, comprising five (5) holes for a total of 284 m into the South Zone.

2008-2009 A drilling program is carried out by Clifton Star, comprising 209 holes (58,053 m) on the Beattie property and 99 holes (37,566 m) on the Donchester property.

2010 Under the terms of a joint venture agreement with Osisko Mining Corp. ("Osisko"), the latter becomes the operator of a drilling program comprising 314 holes for a total of 102,529 m on the Beattie and Donchester properties. Osisko also carries out a channel sampling program consisting of 220 channels (460 m of cut channels) to complement the drilling program.

2010 Osisko contracted SGS Mineral Service (Lakefield, Ontario) to conduct testwork on Beattie Duparquet Project samples. The program included comminution testwork and preliminary cyanidation and flotation tests to investigate the recovery of gold.

2011 During December and January 2011, Geophysics GPR International Inc. flew a helicopter-borne magnetic and time-domain electromagnetic geophysical survey for Osisko Mining Corporation's Duparquet Project. The survey was composed of one (1) single block and covers all Duparquet Project.

2011 Osisko contracted SGS Canada Inc. ("SGS") (Geostat) to prepare a NI 43-101 compliant Mineral Resource Estimate on the Beattie sector only.

2011 Clifton Star continued drilling the Beattie and Donchester properties and starts drilling the Central Duparquet property. A total of 46 holes and 28 holes extensions are drilled for a total of 17,565 m.

2012 Clifton Star completed surface outcrop stripping on and in the vicinity of the RWRS Zone, South Zone and the North Zone. A total of nineteen (19) outcrops on Beattie, nine (9) on Donchester and one (1) on Central Duparquet properties were mechanically stripped and then sampled using a conventional channelling technique. The primary goal for the stripping and channel sampling was to test and verify the gold mineralization continuity up to surface.

2012 Clifton Star decided to resample 50 of the company's previous holes that had been selectively sampled at the time of drilling. The selective nature of the sampling resulted in untested shoulders adjacent to mineralized zones.

2012 From January to the end of August 2012, Clifton Star drilled a total of 35 new holes and eight (8) extensions of older holes. Overall, the drilling program produced 12,471 m of NQ-size core during this period. From September 2012 until January 2013, Clifton Star continued drilling the Duparquet properties. A total of 53 drill holes and ten (10) drill hole extensions were completed during this period, for a total of 22,675 m of NQ-size core.

2012 Clifton Star contracted Tenova Mining & Minerals – Bateman Engineering Pty Ltd ("Tenova-Bateman") to develop preliminary capital and operating costs for the construction and operation of a mineral processing plant to process from the Duparquet Project to produce gold doré. The purpose of the Study was to assess the viability of the Total Pressure Oxidation, Albion Process™ and Biox® Leaching technologies to treat gold rich concentrate from the Duparquet mine deposits, in order to produce gold doré bar and to provide capital and operating cost estimates to a level of accuracy of ±35% for the proposed three flowsheets.

2012 Clifton Star contracted SGS (Lakefield, Ontario) to conduct testwork on Duparquet Project samples. The program included flotation, pressure oxidation and cyanidation testwork to investigate the recovery of gold from ore and tailing samples. Preliminary comminution and environmental tests were also conducted.

2012 In fall, 15 of the 19 holes of the Dumico property were entirely resampled and/or downhole surveyed by Clifton Star. Down-hole orientation surveys were done using a Gyroscope instrument.

2013 Clifton Star sent to SGS a 12 t composite bulk sample of the Duparquet Project mineralized zones, from large diameter drill core, for metallurgical and environmental pilot tests. The planned testwork included a continuous pilot plant test for POX and also for high grade gold concentrates production. Results received in September 2013 confirmed previous recoveries.

2013 Clifton Star began a surface outcrop stripping program on and in the vicinity of the RWRS Zone, South Zone and the North Zone. A total of 41 outcrops on Beattie property were mechanically stripped and then sampled using a conventional channelling technique. The primary goal for the stripping and channel sampling was to test and verify the gold mineralization continuity up to surface. The 2013 channel sampling program was not included in the most recent mineral resource estimate.

2013 Clifton Star decided to resample eight (8) of the company's previous holes that had been selectively sampled at the time of drilling. The selective nature of the sampling resulted in untested shoulders adjacent to mineralized zones. Clifton Star's re-sampling program successfully filled in the gaps in these holes.

2013 A drilling program is carried out by Clifton Star, comprising 92 holes (16,773.5 m) on the Beattie, Donchester and Central Duparquet properties.

6.2 History of resource and reserve estimates

Table 6-2 – Summary of history of resource and reserve estimates for the Duparquet Project

Year	Company	Work
1987	SOQUEM Inc. Cambior Inc.	Reserve estimates
2005	Golder Associates Inc. Beattie Gold Mines Ltd	Resource estimate for Central Duparquet Project
2009	Genivar Inc. Clifton Star Resources Inc.	Unpublished MRE for the tailing ponds (Fillion, 2009)
2011	SGS Canada Inc. (Geostat) Osisko Mining Inc. and Clifton Star Resources Inc.	NI 43-101 MRE on the Beattie Project only

Year	Company	Work
2012	InnovExplo Inc. Clifton Star Resources Inc.	Maiden NI 43-101 MRE for the global Duparquet Project. The MRE included the tailings resource from the 2009 Genivar report
2012	InnovExplo Inc. Clifton Star Resources Inc.	Updated NI 43-101 MRE for the global Duparquet Project (including the Genivar 2009 tailings resource) NI 43-101 PEA
2013	InnovExplo Inc. Clifton Star Resources Inc.	NI 43-101 MRE for the global Duparquet Project (including the 2009 Genivar tailings resource) Dumico DDHs added to this update
2013	InnovExplo Inc. Clifton Star Resources Inc.	NI 43-101 MRE with PEA for the global Duparquet Project (including the Genivar 2009 tailings resource)
2014	InnovExplo Inc. Clifton Star Resources Inc.	NI 43-101 technical report and PFS (including the Genivar 2009 tailings resource)

1987 Reserve estimates of the Beattie mine, as carried out by C.W. Archibald Ltd (1987) and Derry, Michener, Booth & Wahl (1987), appear to be based on a 1950 longitudinal section (see reference in Bevan, 2011).

2005-2007 Golder Associates Inc. performs a reserve estimate in 2005 for the Central Duparquet property.

2009 Genivar Inc. is contracted to produce a Mineral Resource Estimate for a portion of the mine tailings area (Fillion, 2009), but the report was never published.

2011 Osisko contracted SGS to prepare a NI 43-101 compliant Mineral Resource Estimate on the Beattie sector only.

2012 Clifton Star contracted InnovExplo to prepare a NI 43-101 compliant Mineral Resource Estimate on the Duparquet Project, combining the resources of all three (3) adjacent properties (Beattie, Donchester, Central Duparquet) and audit Genivar's Mineral Resource Estimate of the tailing ponds in order to incorporate the results into the "global" mineral resource estimate.

2012 Clifton Star contracted InnovExplo to up-date the NI 43-101 compliant Mineral Resource Estimate and prepare a NI 43-101 compliant Preliminary Economic Assessment (PEA) for the Duparquet Project. The Dumico DDHs were added to this update. The PEA Study was prepared as an open pit mining project relating solely to the mineral resources located on the Duparquet Project.

2013 Clifton Star contracted InnovExplo to up-date the NI 43-101 compliant Mineral Resource Estimate on the Duparquet Project. The Dumico DDHs were added to this update.

2013 Clifton Star contracted InnovExplo to up-date the NI 43 101 compliant Mineral Resource Estimate and a PEA on the Duparquet Project (Beattie, Donchester, Central Duparquet, Dumico).

2014 Clifton Star contracted InnovExplo to up-date the NI 43-101 compliant Mineral Resource Estimate and prepare a NI 43-101 compliant Preliminary Feasibility Study (PFS) for the Duparquet Project.

7. GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

7.1.1 Archean Superior Province

The Archean Superior Province (Figure 7-1) forms the core of the North American continent and is surrounded by provinces of Paleoproterozoic age to the west, north and east, and the Grenville Province of Mesoproterozoic age to the southeast. Tectonic stability has prevailed since ~2.6 Ga in large parts of the Superior Province. Proterozoic and younger activity are limited to rifting of the margins, emplacement of numerous mafic dyke swarms (Buchan and Ernst, 2004), compressional reactivation, large-scale rotation at approximately 1.9 Ga, and failed rifting at ~1.1 Ga. The craton has largely escaped ductile deformation, with the exception of the northwest and northeast Superior Province margins that were pervasively deformed and metamorphosed at 1.9 to 1.8 Ga.

A first-order feature of the Superior Province is its linear subprovinces, or “terranes”, of distinctive lithological and structural character, accentuated by subparallel boundary faults (e.g., Card and Ciesielski, 1986). Trends are generally E-W in the south, WNW-ESE in the northwest, and NW-SE in the northeast. In Figure 7-1, the term “terrane” is used in the sense of a geological domain with a distinct geological history prior to its amalgamation into the Superior Province during the 2.72 Ga to 2.68 Ga assembly events, and a “superterrane” shows evidence for internal amalgamation of terranes prior to the Neoarchean assembly. “Domains” are defined as distinct regions within a terrane or superterrane.

The Project is located within the Abitibi terrane. This terrane hosts some of the richest mineral deposits of the Superior Province (Figure 7-1), including the large gold camps of Ontario and Quebec (Robert and Poulsen, 1997; Poulsen et al., 2000).

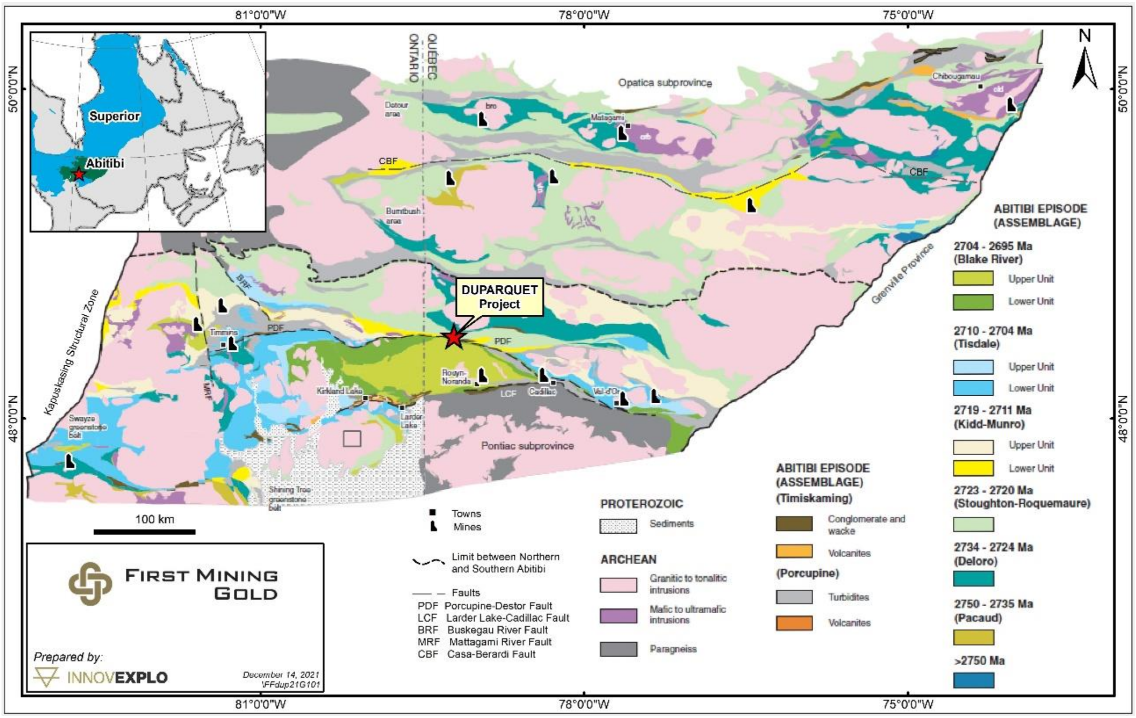


Figure 7-1 – Geological map of the Superior Province

7.1.2 The Abitibi Terrane (Abitibi Subprovince)

The Abitibi Subprovince (Abitibi Greenstone Belt) is located in the southern portion of the Superior Province (Figure 7-1). The Abitibi Subprovince is divided into the Southern and Northern volcanic zones ("SVZ" and "NVZ"; Chown et al. 1992), representing a collage of two arcs delineated by the Destor-Porcupine-Manneville Fault Zone ("DPMFZ"; Mueller et al. 1996). The SVZ is separated from the Pontiac Terrane sedimentary rocks, an accretionary prism (Calvert and Ludden 1999) to the south, by the Cadillac–Larder Lake Fault Zone ("CLLFZ"). The fault zones are terrane "zippers" that display the change from thrusting to transcurrent motion as documented in the turbiditic flysch basins unconformably overlain by, or in structural contact with, coarse clastic deposits in strike-slip basins (Mueller et al. 1991, 1994, 1996; Daigneault et al. 2002). A further subdivision of the NVZ into external and internal segments is warranted, based on distinct structural patterns with the intra-arc Chicobi sedimentary sequence representing the line of demarcation. Dimroth et al. (1982, 1983a) recognized this difference and used it to define internal and external zones of the Abitibi greenstone belt. Subsequently, numerous alternative Abitibi divisions were proposed (see Chown et al., 1992), but all models revolved around a plate tectonic theme. The identification of a remnant Archean north-dipping subduction zone by Calvert et al. (1995) corroborated these early studies.

The 2735-2705 Ma NVZ is ten times larger than the 2715-2697 Ma SVZ. Both granitoid bodies and layered complexes are abundant in the former. In contrast, plume-generated komatiites, a distinct feature of the SVZ, are only a minor component of the NVZ, observed only in the Cartwright Hills and Lake Abitibi area (Daigneault et al. 2004). Komatiites rarely constitute more than 5% of greenstone sequences and the Abitibi is no exception (Sproule et al. 2002). The linear sedimentary basins are significant in the history because they link arcs and best chronicle the structural evolution and tempo of Archean accretionary processes. The NVZ is composed of volcanics cycles 1 and 2, which are synchronous with sedimentary cycles 1 and 2, whereas the SVZ exhibits volcanic cycles 2 and 3, with sedimentary cycles 3 and 4 (Mueller et al. 1989; Chown et al. 1992; Mueller and Donaldson 1992; Mueller et al. 1996).

The Abitibi Subprovince displays a prominent E-W structural trend resulting from regional E-trending folds with an axial-planar schistosity that is characteristic of the Abitibi belt (Daigneault et al. 2002). The schistosity displays local variations in strike and dip, which are attributed to either oblique faults cutting the regional trend or deformation aureoles around resistant plutonic suites. Although dominant steeply-dipping fabrics are prevalent in the Abitibi Subprovince, shallow-dipping fabrics are recorded in the Pontiac Subprovince and at the SVZ-NVZ interface in the Preissac-Lacorne area. The metamorphic grade in the Abitibi Subprovince displays greenschist to sub-greenschist facies (Joly, 1978; Powell et al., 1993; Dimroth et al., 1983b; Benn et al., 1994) except around plutons where amphibolite grade prevails (Joly, 1978). In contrast, two extensive high-grade zones coincide with areas of shallow-dipping fabrics: (1) the turbiditic sandstone and mudstone of the Pontiac Subprovince at the SVZ contact, which exhibit a staurolite-garnet-hornblende-biotite assemblage (Joly, 1978; Benn et al., 1994); and (2) the Lac Caste Formation turbidites at the SVZ-NVZ interface (Malartic segment) with sandstone and mudstone metamorphosed to biotite schist with garnet and staurolite. Feng and Kerrich (1992) suggested that the juxtaposition of greenschist and amphibolite grade domains indicates that uplift occurred during the compressional stage of collisional tectonics.

7.2 Local Geological Setting

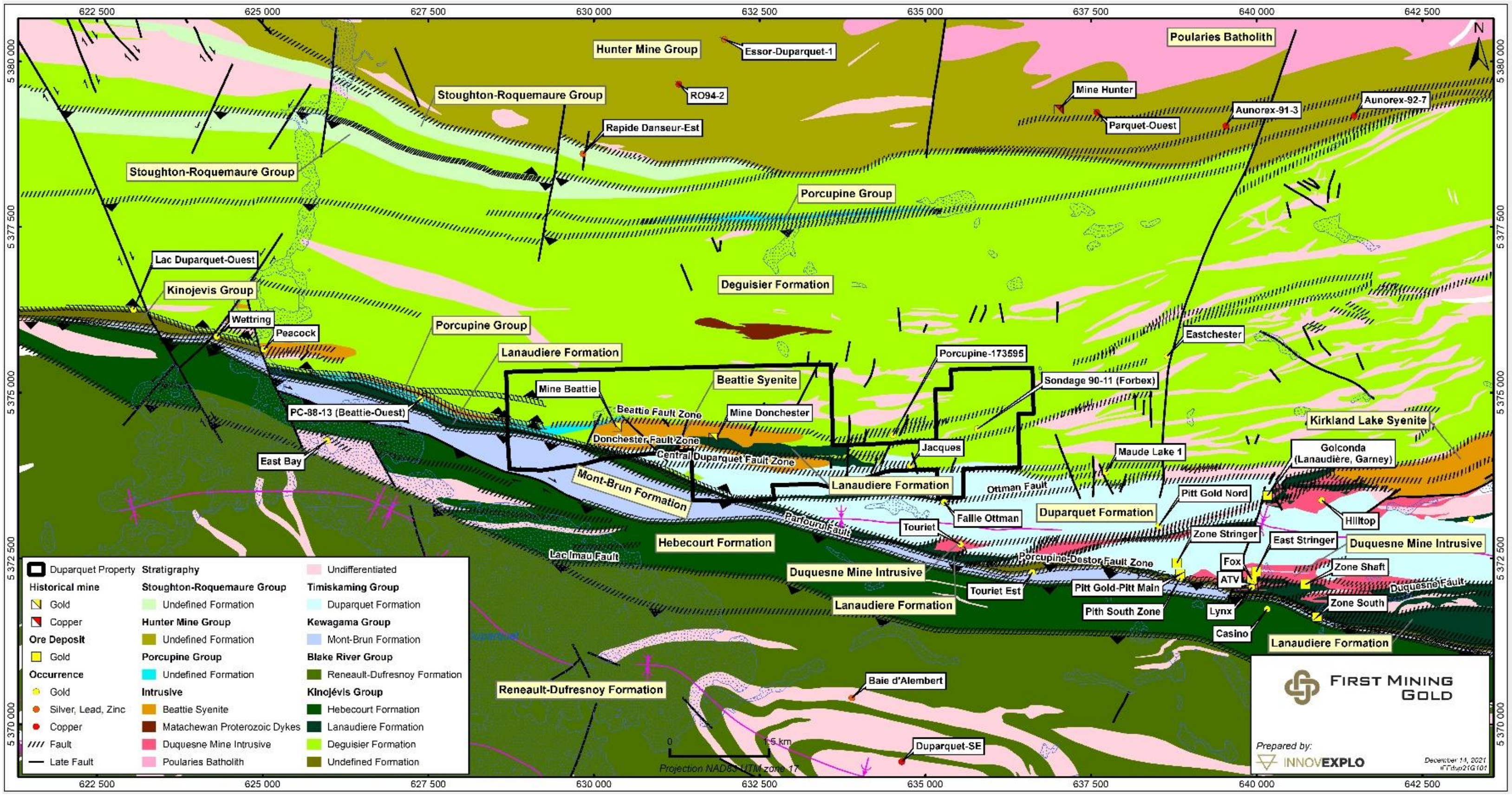
The local geological setting and Project geology are represented by the Kinojevis, Timiskaming and Blake River groups (Figure 7-2). The Blake River Group is south of the DPMFZ, characterized by a 2706-2696 Ma volcanic sequence, 4 to 7 km thick, belonging to the SVZ. It comprises mafic volcanic rocks with several felsic volcanic centres interpreted to be a mega-caldera complex representing a multi-stage collapse structure occupying most of the present-day surface area of the Blake River Group (Pearson and Daigneault, 2009). Several arguments support a mega-caldera complex, including: (1) the mafic-intermediate swarm pattern; (2) the overall geometry; (3) the fault pattern; (4) the distribution of volcaniclastic rocks; and (5) the distribution pattern of carbonate-rich hydrothermal alteration (Pearson and Daigneault, 2009).

The Kinojevis Group is north of the DPMFZ and subdivided into two units: the Deguisier Formation, composed of massive to pillowed tholeiitic basalt associated with some amounts of andesite, felsic pyroclastic rocks and gabbro, and the Lanaudière Formation, consisting of basalts, andesites, rhyolites, komatiites and multiple mafic to ultramafic intrusions (Goutier and Lacroix, 1992).

The Duparquet Formation, part of the Timiskaming Group, is a sedimentary unit within a small structurally controlled basin known as the Duparquet Basin (Mueller et al., 1991). The basin's development at a late orogenic stage (of the Kenoran orogeny) classifies it as a successor basin (pull-apart basin). This wedge-shaped basin, delineated by bounding faults, may be compared to divergent fault-wedge basins in the Cenozoic basins of southern California (Crowell, 1974). Mueller et al. (1991) suggested a late

Archean dextral strike-slip movement within a regime dominated by north-south compression.

Detailed stratigraphic mapping in the Duparquet Formation led to the recognition of three mutually transitional facies associations (Mueller et al., 1991). Based on predominant lithology, grain size and sedimentary structures, the facies end members are as follows: (1) conglomerate-sandstone facies association (“CSFA”); (2) sandstone-argillite facies association (SAFA); and (3) argillite-sandstone facies association (“ASFA”). The CFSA, up to 100 m thick, predominates near-faulted basin margins and can be divided into sub-facies in which the porphyry clast component is either dominant or negligible. The SAFA is characterized by: a set of coarse to very coarse-grained trough cross-beds 5 to 100 cm thick; planar to wavy, medium to coarse-grained beds 5 to 20 cm thick; and argillite beds 1 to 30 cm thick. The ASFA is distinguished by laterally continuous, well-laminated argillite interstratified with fine to coarse-grained sandstone beds 2 to 10 cm thick.



Modified from SIGEOM

Figure 7-2 – Regional geological setting of the Duparquet Project

7.3 Duparquet Project Geological Setting

This section is a slightly modified version of the project geology description provided in the technical report by Bevan (2011). The authors have reviewed and compared Bevan's geological description to other such accounts in publicly available documents and consider it accurate to the best of their knowledge.

The Project straddles syenitic plutons and the Kinojevis, Duparquet and Mont-Brun formations (Figure 7-3). The Project area is characterized by the presence of two syenitic plutons oriented E-W. These intrusions are bounded by E-W major faults, which are interpreted as splays of the main SE-trending DPMFZ, which clips the southwest corner of the Project. The geological formations generally strike E-W and dip steeply (80°-85°) to the north. The metamorphic grade is low (greenschist facies), and local alteration is represented by chloritization, silicification and sericitization. Most of the known mineralization appears to be related to late intrusions of syenite and feldspar porphyry in the Keewatin mafic flows and tuffs, along zones of weakness adjacent to or coincident with the E-W major faults.

7.3.1 Stratigraphy

The rocks underlying the Project are generally made up of intercalated felsic (rhyolitic to dacitic) and mafic (basaltic to andesitic) metavolcanic flows, with the felsic flows being the oldest. Metasedimentary layers are also present, consisting of arkosic sandstones, greywackes, argillites, crystal tuffs and conglomerates, and are generally more prevalent on the south side of the DPMFZ. All the units have been intruded by syenite porphyry units, which appear to be concordant with the location of the major fault zones. Quartz feldspar and lath porphyries were injected along minor faults affecting the syenite intrusions.

7.3.2 Structural Geology

The predominant structures on the Project are the E-W splays of the DPMFZ. The Beattie Fault Zone ("BFZ") is located along the north contact of the main syenite body, whereas the Donchester Fault Zone ("DFZ") is located along the south contact. The Central Duparquet Fault Zone ("CDFZ") is located along the south contact of the second smaller syenite intrusive to the east. The syenite porphyry generally plunges to the east.

The Beattie and Donchester fault zones dip steeply to the south and north, respectively, suggesting that the contacts of the syenite porphyry converge within the central portion of the complex at depth. The CDFZ has an orientation subparallel to the DFZ. Strike-slip offset along these major structures is not significant. Neither the downdip component of movement nor the slip vector could be determined.

Late crosscutting faults interpreted in plan view (Figure 7-3) are likely to have some impact on displacing the gold-bearing zones. However, the authors were unable to verify their existence while modelling the mineralized solids. Any displacement along such crosscutting faults is assumed to be minimal and without significant consequences at the scale of InnovExplo's interpretation. This is supported by the small amount of horizontal displacement visible on the drift maps and mine level drilling plans.

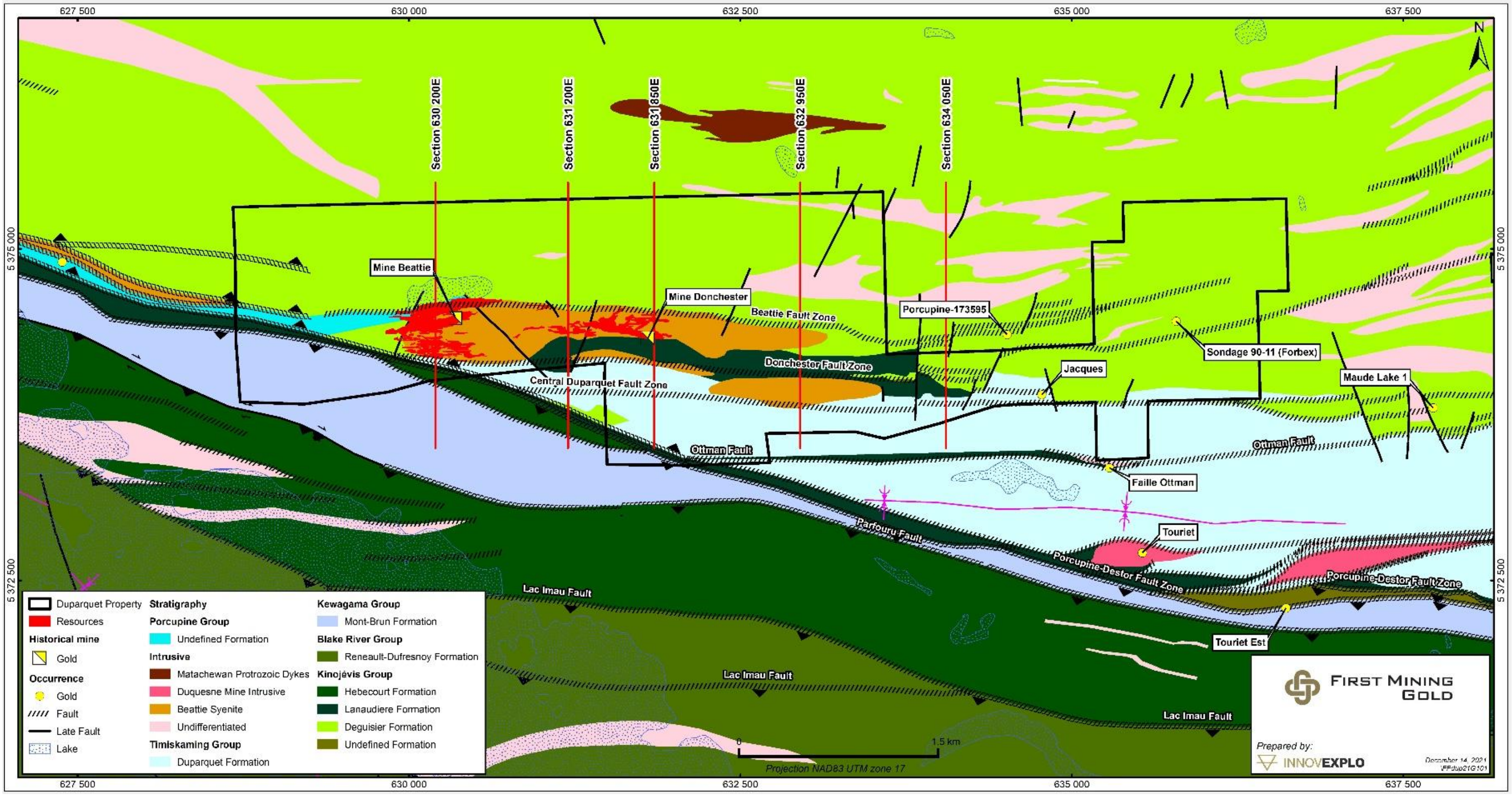


Figure 7-3 – Geology of the Duparquet Project. Structural and geometric details are represented by surface projections of the relevant structural elements from the deposit-scale 3D litho-structural model.

7.3.3 Alteration

Gold-bearing quartz veins within the DPMFZ have historically been associated with sericite-carbonate-ankerite-chlorite alteration haloes and late-stage quartz-carbonate veins with ankerite haloes.

Gold-bearing mineralization within the Duparquet deposit (including the Beattie mine, Donchester mine and Central Duparquet deposit) is associated with carbonate, chlorite, fuchsite and sericite alteration as a product of hydrothermal fluid injection within sheared and brecciated sections of the syenite porphyries. Silicification and chert-calcite-rich accumulations along fractures have been observed within the mineralized zones, accompanied by pyrite-arsenopyrite host rock replacement. The chert is dark grey due to its potassium-rich composition and the hematite and tourmaline content of the hydrothermal fluids.

7.3.4 Mineralization

Gold mineralization at the Beattie mine has been historically associated with silicified and brecciated zones containing a low percentage of very fine-grained pyrite and arsenopyrite (Goutier and Lacroix, 1992). According to Bevan (2011), the “main” type of gold mineralization in the Duparquet deposit generally occurs within shears or brecciated zones along or within the adjacent intrusive syenitic masses and is associated with finely disseminated pyrite and minor arsenopyrite replacement. Sulphide content is generally low (0.5 to 4%), although it can be up to 10% in some cases. Higher gold grades appear to be related to the finer-grained sulphides (Bevan, 2011). Historically, gold production at the Beattie mine was accompanied by the extraction of arsenic trioxide and silver as by-products. The “breccia” type of mineralized material is found within the metavolcanic rocks (volcanics and tuffs) and consists of well-mineralized, siliceous, brecciated, grey-coloured and bleached zones. The porphyry-type mineralized material consists of fine-grained and strongly silicified mineralized zones hosted by porphyry intrusives. They generally have lower gold grades than other types of mineralized zones within the deposit (Bevan, 2011).

The typical mineral assemblage in mineralized zones of all types is characterized by feldspar, quartz, sulphides (pyrite and arsenopyrite), sericite, chlorite and other secondary minerals. Mill tests suggest that some 35% of the gold is in a free state, with the remainder associated with sulphides. According to Bevan (2011), three phases of gold enrichment or remobilization can be interpreted from the cross-cutting relationships between gold-bearing veins. Bevan (2011) also states that higher gold contents are found along cross-cutting faults, in fold noses, and within the lath-textured porphyry dyke intrusion as a consequence of the remobilization processes.

At the Beattie mine, the main mineralized lens is hosted by a shear zone (the BFZ) at the northern contact of the syenite intrusion (Figure 7-4). In this report, the main zone is referred to as the North Zone. A second gold-bearing lens, also hosted by a shear zone (the DFZ) but occurring at the south contact of the syenite body, is also known at the Beattie mine and is referred to as the South Zone herein. Gold mineralization at the Donchester mine was of higher grade and associated with an E-W shear zone cutting across some volcanic rocks and syenitic dykes (Goutier and Lacroix, 1992). This zone is interpreted herein as the east extension of the South Zone. At both the Beattie and Donchester mines, the South Zone can be subdivided into several mineralized “lenses”, modelled herein as ten (10) individual subzones. Six (6) other major striking mineralized zones occurring within the Beattie-Donchester area have been interpreted by InnovExplo for the purpose of the current mineral resource estimate.

Mineralization on the Central Duparquet property is hosted by the CDFZ and is of a similar nature as the South and North zones (Bevan, 2011). InnovExplo interpreted three (3) mineralized zones at Central Duparquet.

The Dumico property is the eastern extension of the Central Duparquet property. InnovExplo interpreted five (5) mineralized zones at Dumico. Three of these strike E-W and are interpreted as the extensions of the CD Zones found at Central Duparquet. The other two zones, which strike NW-SE, occur on the eastern portion of the Dumico Project. Based on the current interpretation, they are thought to be associated with a subsidiary structure subparallel to the regional DPMFZ.

Thirty-four (34) secondary mineralized zones have been interpreted within the previously defined “inter-zone” mineralized envelope. The interpretation of these secondary mineralized zones, most of which strike SW-NE, is based on field observations and grade continuity throughout the sample point dataset. These zones are interpreted to be hosted by subsidiary structures associated with the BFZ and DPFMZ.

The geometry, size and structural context of all the zones discussed above are shown in Figure 7-4 to Figure 7-8. The interpretation displays continuous mineralization over a 4.5-km strike-length corridor that measures 1 km wide and extends to about 1 km below the surface. This mineralized corridor contains a relatively complex system of E-W and SW-NE striking structures hosting mineralized lenses, mainly confined between the BFZ and DFZ. Additional details on InnovExplo’s interpretation of the mineralized zones on the Project can be found in Section 14.3 - Geological Model.

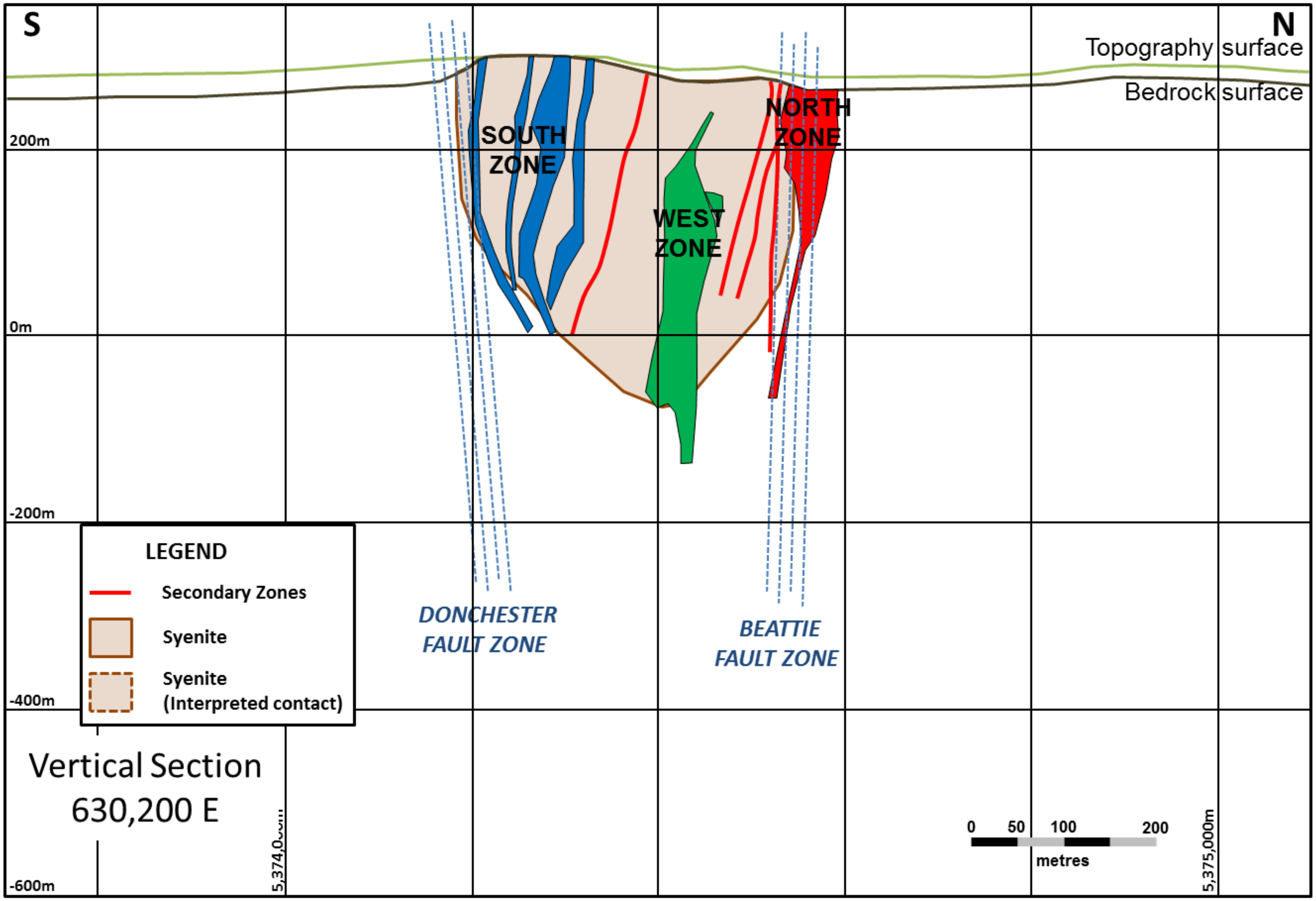


Figure 7-4 – 3D litho-structural model. Vertical section 630,200E, looking west (location on Figure 7-3)

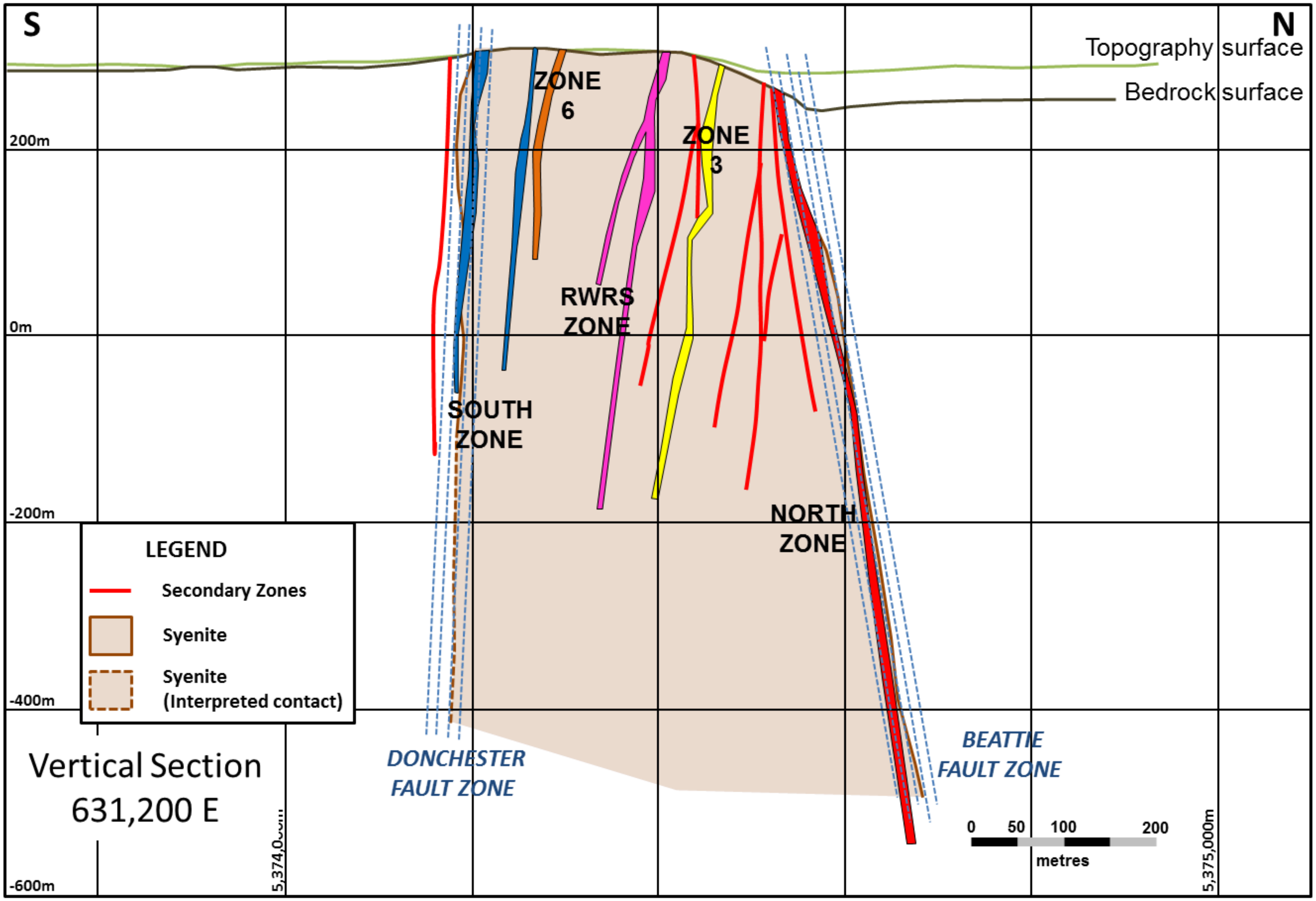


Figure 7-5 – 3D litho-structural model. Vertical section 631,200E, looking west (location on Figure 7-3)

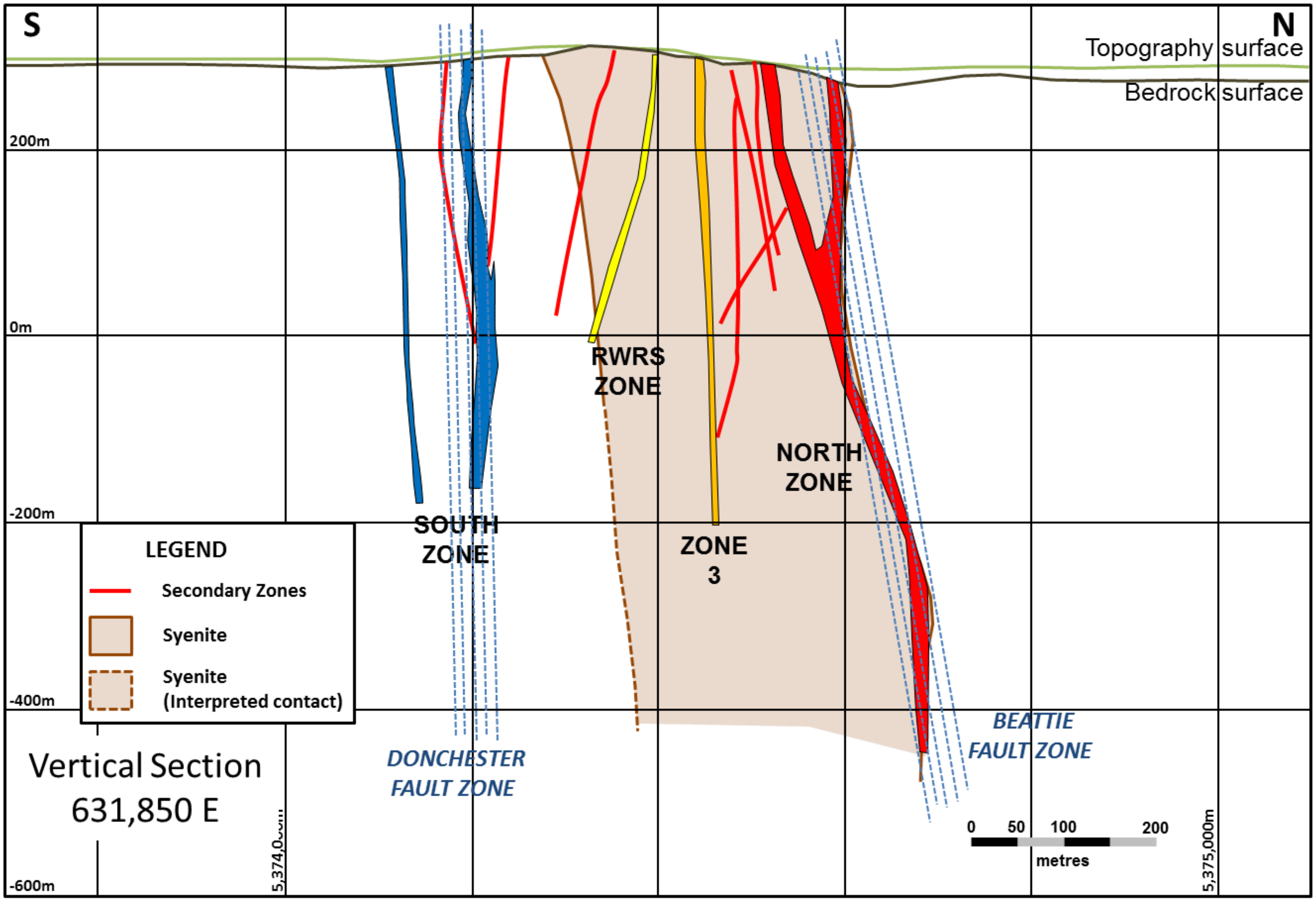


Figure 7-6 – 3D litho-structural model. Vertical section 631,850E, looking west (location on Figure 7-3)

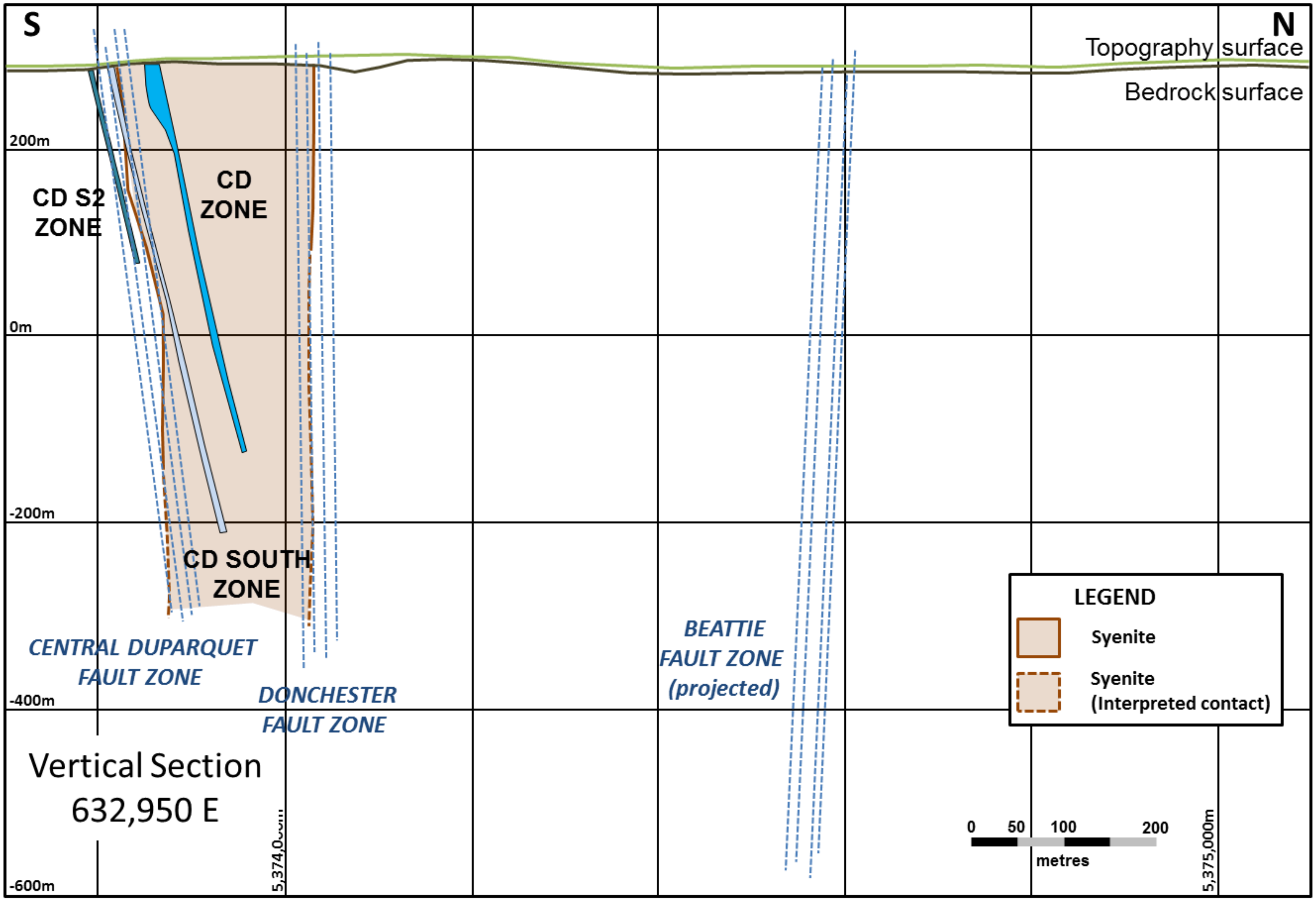


Figure 7-7 – 3D litho-structural model. Vertical section 632,950E, looking west (location on Figure 7-3)

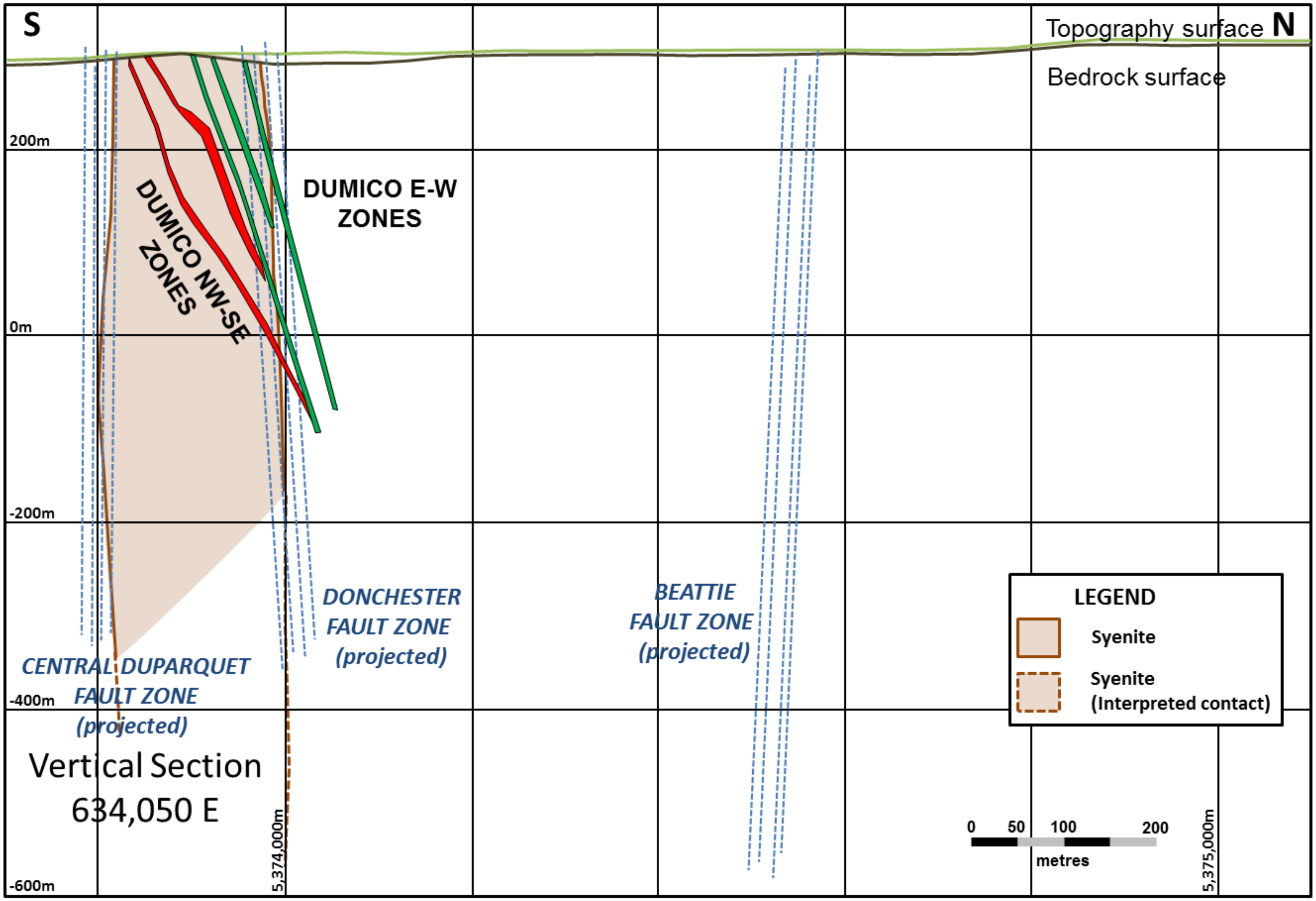


Figure 7-8 – 3D litho-structural model. Vertical section 634,050E, looking west (location on Figure 7-3)

8. DEPOSIT TYPES

The standard orogenic gold model characterizes the majority of gold deposits within the Abitibi belt. However, several examples of late mineralization are associated with alkaline intrusions (Robert, 2001), thus differing from the standard orogenic gold model. Syenite-associated disseminated gold deposits in the Abitibi greenstone belt consist of zones of disseminated sulphides with variably developed quartz stockworks, which are intimately associated with Timiskaming-age, monzonitic to syenitic porphyry intrusions (Robert, 2001). Like quartz-carbonate vein deposits, all known syenite-associated disseminated gold deposits in the southern Abitibi belt occur along a major fault. As a result of their distribution along major faults, these deposits commonly occur at or near boundaries between contrasting lithological domains. Examples of these deposits are Young-Davidson, Matachewan Consolidated, Ross, Holt-McDermott, and Lightening in Ontario; and Beattie, Douay, Canadian Malartic, East Malartic, and Barnat-Sladen in Quebec.

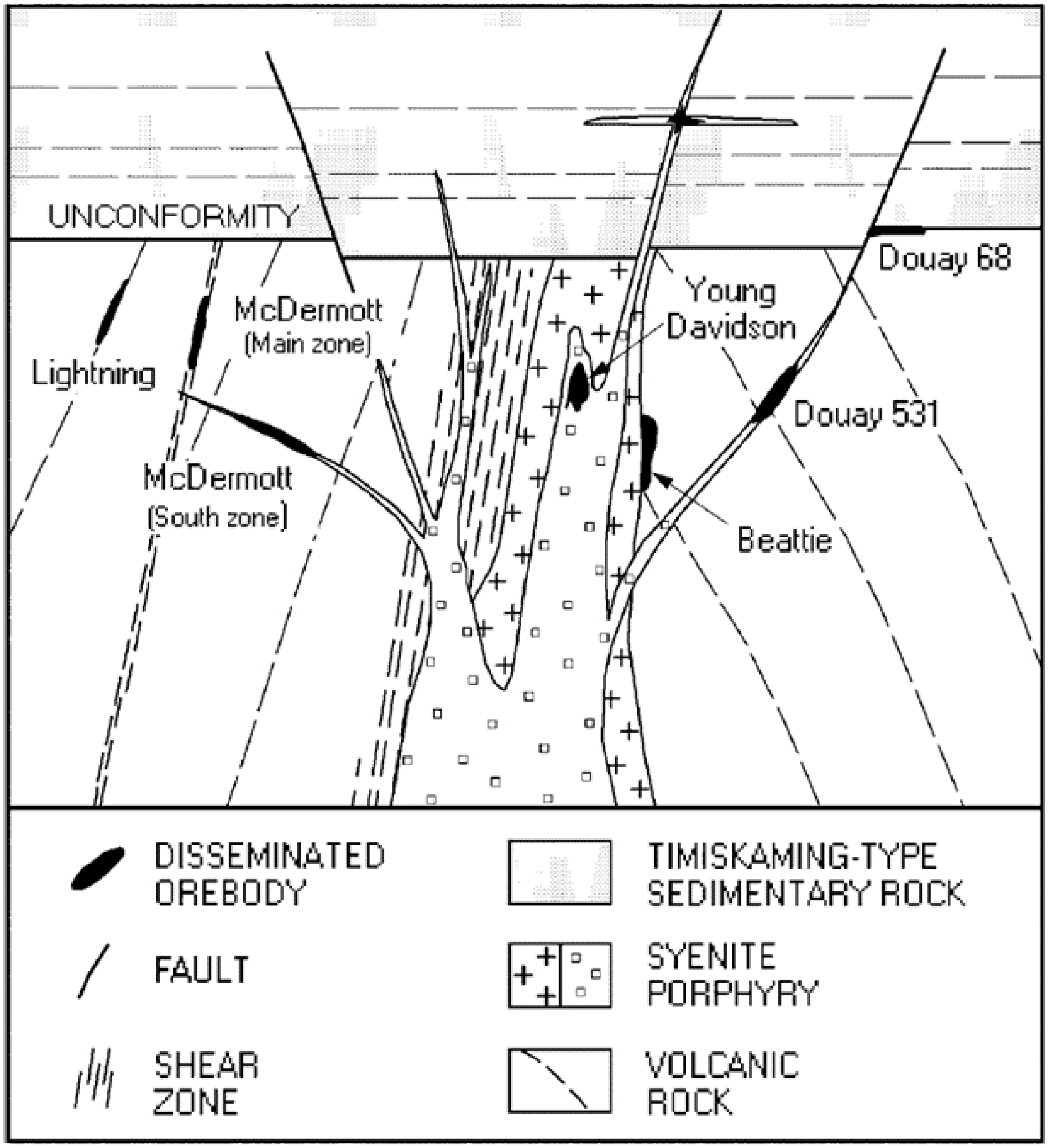
The Timiskaming-type sedimentary rocks occur along restricted segments of major fault zones, where they are preserved as synclinal keels (Muller et al., 1991). The syenite intrusions form small stocks, commonly elongated subparallel to the overall structural trend and generally surrounded by numerous satellite dykes (Robert, 2001). Although some intrusive phases are equigranular, most are porphyritic, with K-feldspar phenocrysts in a fine-grained to aphanitic groundmass. Both Timiskaming-type sedimentary rocks and syenitic intrusions have been overprinted by at least one generation of structural fabrics and folds (Corfu et al., 1991). The sedimentary rocks are folded into tight synclinal structures, which probably accounts for their preservation, subparallel to the trace of the major faults along which they occur (Muller et al., 1991). A related penetrative foliation, best developed in sedimentary rocks, parallels the regional penetrative E-W, subvertical S_2 foliation. This foliation is absent or only weakly developed in the larger syenitic intrusions, except where they have been sericitized. Overprinted faults and shear zones are also common in these deposits (Robert, 2001). They range from relatively ductile shear zones to narrow brittle faults. The older faults are parallel to the regional foliation overprinting the Timiskaming-type sedimentary rocks.

Syenite-associated disseminated gold deposits consist of zones of disseminated sulphides with variably developed stockworks in intensely altered wallrocks (Robert, 2001). They have sharp diffuse limits, defined by a decrease in sulphide content, gold grades, and intensity of stockwork fracturing. Owing to the abundance of micro-veinlet stockworking and fracturing, many orebodies take on a breccia appearance. The morphology of the deposits ranges from overall tabular to pipe-like, although many have rather irregular outlines. Most mineralized zones are steeply dipping or steeply plunging, but examples of moderately to shallowly dipping orebodies, discordant to lithological units, are also known (Robert, 2001).

The total sulphide mineral content of the orebodies is typically less than 10% by volume, and commonly only a few percent (Robert, 2001). Disseminated sulphides are fine- to very fine-grained and consist dominantly of pyrite, with significant arsenopyrite in a few deposits. Associated stockworks consist of millimetre- to centimetre-thick veinlets of grey to cherty quartz, with subordinate amounts of carbonate (Fe-dolomite and calcite), albite, and pyrite. In addition to pyrite and arsenopyrite, ore-related metallic minerals include minor to trace amounts of chalcopyrite and hematite. Telluride minerals, molybdenite, and magnetite are common associates of this type of mineralization, whereas galena, tennantite and bismuthinite occur at few deposits. Accordingly, orebodies are generally enriched in Cu, As, Te, with common but variable enrichments in Pb, Mo, W, Zn, and locally Sb. The gold-to-silver ratios of the ores generally range from about 1:1 to 5:1.

Zones of hydrothermal alteration are spatially coincident with zones of disseminated sulphide minerals and veinlet stockworks. The most intense alteration corresponds in a general way to economic mineralization (Robert, 2001). Carbonatization and albitization are significant alteration types at nearly all deposits; K-feldspar alteration and sericitization are also present in several deposits.

The disseminated gold orebodies occur in a variety of positions relative to syenite intrusions, ranging from proximal to distal (Robert, 2001). A conceptual model is proposed by Robert (2001) for the occurrence of disseminated gold mineralization as part of large, syenite-centred hydrothermal systems (Figure 8-1). This system is shown in its interpreted primary configuration prior to post-Timiskaming folding and faulting in which a composite syenite stock intrudes a pre-Timiskaming fault zone juxtaposing contrasting lithological domains. The early intrusive phase is truncated by the unconformity, whereas the younger phase intrudes sedimentary rocks.



From Robert, 2001

Figure 8-1 – Schematic geological model showing the distribution of disseminated-stockwork orebodies relative to a composite syenite stock intruded along a major fault zone, near the base of the Timiskaming unconformity

The Beattie Syenite is an Archean porphyry intrusion emplaced along the DPMFZ (Figure 7-3). The syenite is aligned along an E-W axis. It is hosted by mafic and intermediate volcanic rocks and is penecontemporaneous with the Timiskaming sedimentary rocks deposition. The main gold mineralization is associated with a network of E-W dextral strike-slip faults dipping steeply to the north. Syenitic intrusions indirectly control the location of gold deposits by their effect on the development of potentially mineralized structures during diapirism. The current structures hosting the mineralization are shear zones and folds. The Duparquet Formation is characterized by folding (Goutier and Lacroix, 1992). Gold deposits in the Duparquet area are genetically related to these intrusive rocks. The gold deposits tend to occur in concentric, normal and radial strike-slip faults and shears. Lateral ballooning and forceful intrusion may also generate concentric compressional structures (reverse and strike-slip faults and compressional fabrics). Gold deposits in the Duparquet area tend to occur close to the syenite-sedimentary and/or volcanic rock contacts. This is attributed to the competency contrasts between the syenitic rocks and the Duparquet Formation lithologies during deformation, resulting in favourable structural traps for gold mineralization.

In the Duparquet area, the main alteration observed is silicification. Sericitization, ankeritization and chloritization are other types of alteration associated with gold mineralization (Goutier and Lacroix, 1992). The main sulphide is pyrite (<10%), accompanied by arsenopyrite (Davidson and Banfield, 1944). Gold is hosted in arsenopyrite and arsenian pyrite (Bigot and Jébrack, 2012). Gold grains are less than micron-sized. The association with arsenian minerals and their very small size suggests that gold was incorporated into the crystalline structure of arsenopyrite and arsenian pyrite in solid solution or as nanoparticles (Bigot and Jébrack, 2012).

The metallic assemblage in Beattie syenite is polyphased (Bigot and Jébrack, 2012): (1) a primary phase enriched in iron-titanium appears to have produced martite in a more oxidizing environment; (2) several subsequent sulphidation phases were marked by the presence of pyrite and arsenopyrite, some rich in gold, suggesting crystallization under more reducing conditions and at lower temperatures. During the sulphidation phases, three generations of pyrite are identified; the first generation is arsenian and gold-bearing, whereas the second and third are arsenic-poor and gold-free. A late silica-enriched hydrothermal phase remobilized the gold and is marked by cataclasis. Gold migrated into the fractures developed in the cataclastic pyrite, where it crystallized with silver in the form of electrum.

According to Bigot and Jébrack (2012), several petrological characteristics in the Beattie gold deposit, including gold appearances, metallic mineralogy, type of alterations, and ore control, suggest a shallow magmatic deposit.

9. EXPLORATION

The issuer has not performed exploration work on the Project.

10. DRILLING

The issuer has not carried out drilling programs on the Project.

Since February 28, 2013, the database close-out date for the last mineral resource estimate (Poirier et al., 2014), 57 holes (9,548 m) have been drilled on the Project by Clifton Star, Beattie Gold Mines, and 2588111.

10.1 Drilling Methodology

Foramex Drilling of Rouyn-Noranda carried out the drilling campaigns in 2013 and 2014, Rouillier Drilling of Amos in 2015 and 2016, and Multi Drilling of Rouyn-Noranda in 2017 and 2018.

Collar locations were determined using a handheld GPS. Core size was NQ. Down-hole orientation surveys were performed using a Reflex-Shot instrument.

At the drill rig, the drill helpers laid out the core in core boxes and marked off each 3-m drill run using a labelled wooden block.

10.2 Collar Surveys

Casings were left in place with an identification tag. Collars of drill holes completed by Clifton Star and not included in the previous resource estimate were surveyed in 2013 by Patrick Descarreaux Arpenteur-Géomètre Inc. of La Sarre. Collars of drill holes completed by Beattie Gold Mines, and 2588111 were all surveyed in late 2021 by Patrick Descarreaux Arpenteur-Géomètre Inc.

10.3 Logging Procedures

Clifton Star, Beattie Gold Mines and 2588111 used the facilities at the Beattie mine site for core handling, core logging and storage.

The boxes were opened by company employees at the core shack, and the core was measured for recovery and RQD. A geologist recorded all significant data, including rock type, mineralization, alteration, structures, and textures of interest.

After marking sample intervals on the core, the boxes were transferred to the core cutting room, where a technician sawed the core samples in two equal halves. The sample lengths varied from typically 1.0 m in the mineralized sections to 1.5 m in the more barren sections. Once all sample intervals were sawed, the core technician placed one-half of the core in a labelled sample bag. The sampler stapled the sample tags to the core box underneath the half-core and re-wrote the sample interval's marks and the sample numbers on the remaining half with a grease pencil. Bagged samples were loaded into rice bags, which were labelled with the contained sample intervals and contact information (laboratory and company). The shipment information was entered into the shipment database, and the boxes were transferred to the long-term core farm.

10.4 Drill Programs

Figure 10-1 shows the location of the holes drilled between May 6, 2013, and June 17, 2018, by Clifton Star, Beattie Gold Mines and 2588111.

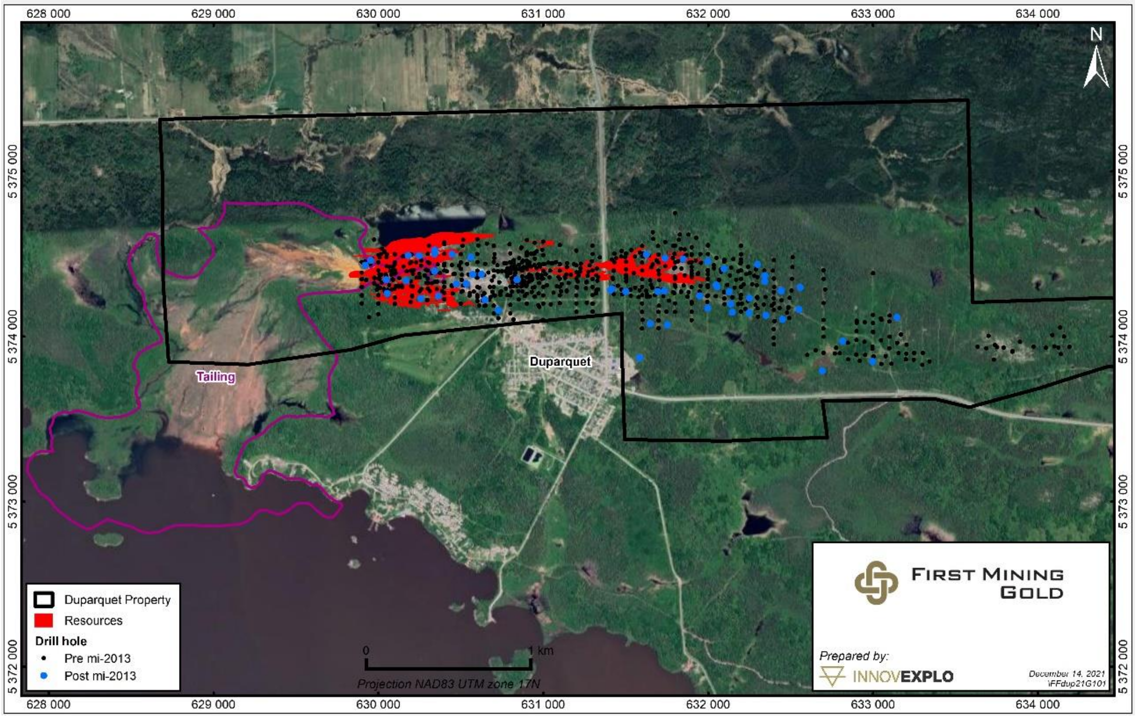


Figure 10-1 – Drillhole locations

10.4.1 2013 Drill Program (Clifton Star)

Clifton Star's 2013 drilling program ran from May 6, 2013, to August 2013. Forty-seven (47) holes were drilled on the Project during this period for 7,422 m of NQ-sized core.

Table 10-1 summarizes the details of the second phase of this drilling program.

Table 10-2 provides the most significant results from this phase (source: Clifton Star press releases of June 19, 2013; August 21, 2013; September 4, 2013).

Table 10-1 – 2013 drilling program, Phase 2

2013 Program	Project			Total
	Beattie	Donchester	Central Duparquet	
No. of DDH	19	22	6	47
Length (m)	2,957	3,648	817	7,422

Table 10-2 – Significant results from the 2013 drilling program, Phase 2

Hole ID	Property	From (m)	To (m)	Length (m)	Grade (Au g/t)	Section (Easting)
BD13-22	Beattie	62.0	68.0	6.0	1.07	630350
BD13-22	Beattie	137.0	171.0	34.0	5.64	630350
BD13-23	Beattie	296.0	314.5	18.5	2.46	630340
BD13-25	Beattie	51.0	68.0	17.0	2.5	630175
BD13-26	Beattie	36.0	43.5	7.5	2.18	629950
BD13-27	Beattie	61.5	133.5	72.0	1.14	630175
BD13-29	Beattie	182.0	201.0	19.0	1.21	630050
BD13-34	Beattie	50.0	61.0	11.0	1.9	630725
BD13-34	Beattie	85.0	99.0	15.0	2.27	630725
BD13-35	Beattie	4.6	10.0	5.4	2.51	630650
BD13-37	Beattie	74.0	108.0	34.0	2.25	629900
BD13-38	Beattie	47.0	86.0	39.0	1.51	630250
BD13-39	Beattie	25.5	37.5	12.0	1.04	630365
CD13-12	Central Duparquet	78.0	84.0	6.0	2.16	632815
D13-15	Donchester	161.1	167.0	5.9	1.66	632450
D13-18	Donchester	145.0	150.0	5.0	5.58	632050
D13-19	Donchester	72.5	77.5	5.0	1.7	631850
D13-20	Donchester	39.0	46.5	7.5	1.51	631625
D13-20	Donchester	81.2	103.1	21.9	1.55	631625
D13-21	Donchester	84.5	100.1	15.6	2.22	631740
D13-25	Donchester	33.0	39.0	6.0	1.84	632150
D13-27	Donchester	24.0	67.5	43.5	1.31	632100
D13-35	Donchester	208.5	214.5	6.0	1.37	632140

10.4.2 2014 to 2018 drilling programs, Beattie Project (Beattie Gold Mines)

Between 2014 and 2018, Beattie Gold Mines drilled five (5) holes on the Beattie Project, at a rate of one hole per year, for a total of 1,088 m. The main purpose of this was drilling was to meet minimum annual expenditures required to keep the Mining Concession in good standing.

Table 10-3 summarizes the details of these holes, and Table 10-4 provides the most significant results.

Table 10-3 – 2014 to 2018 drilling on the Beattie Project

Year	No. DDH	Length (m)
2014	1	213
2015	1	165
2016	1	230
2017	1	240
2018	1	240
Total	5	1,088

Table 10-4 – Significant results from the 2014 to 2018 drill holes on the Beattie Project

Hole ID	From (m)	To (m)	Length (m)	Grade (Au g/t)	Zone (Rock code)
BD16-01	36.0	58.6	22.6	1.48	1500
BD17-01	190.6	204.7	14.1	0.92	1260
BD17-01	44.0	67.7	23.7	0.92	1250
BD14-01	170.9	213.0	42.1	0.66	1100

10.4.3 2014 to 2018 drilling programs, Donchester Project (2588111)

Between 2014 and 2018, 2588111 drilled five (5) holes on the Donchester Project, at a rate of one hole per year, for a total of 1,039 m. The main purpose of this was drilling was to meet minimum annual expenditures required to keep the Mining Concession in good standing.

Table 10-5 summarizes the details of these holes, and Table 10-6 provides the most significant results.

Table 10-5 –2014 to 2018 drilling on the Donchester Project

Year	No. DDH	Length (m)
2014	1	215
2015	1	210
2016	1	210
2017	1	203
2018	1	201
Total	5	1,039

Table 10-6 – Significant results from the 2014 to 2018 drill holes on the Donchester Project

Hole ID	From (m)	To (m)	Length (m)	Grade (Au g/t)	Zone (Rock code)
DON14-01	99.0	103.3	4.3	1.15	1270
DON14-01	66.6	76.4	9.8	0.74	1260
DON18-01	5.3	9.1	3.8	0.72	1250

11. SAMPLE PREPARATION, ANALYSES AND SECURITY

The following paragraphs describe the sample preparation, analyses, and security procedures during the drilling programs carried out between May 6, 2013, and end of 2018, on the Project.

11.1 Core Handling, Sampling and Security

The drill core is boxed, covered and sealed at the drill rigs, and transported by the drilling company employees to the core logging facility at the Beattie mine, where personnel take over the core handling.

The core is logged and sampled by (or under the supervision of) geologists, all of whom are members in good standing of the OGQ (Quebec's professional order of geologists). A geologist marks the samples by placing a unique identification tag at the end of each core sample interval. Sample contacts respect lithological contacts and/or changes in the appearance of mineralization or alteration (type and/or strength). A technician saws each marked sample in half. One half of the core is placed in a plastic bag along with a detached portion of the unique bar-coded sample tag, and the other half is returned to the core box with the remaining tag portion stapled in place. The core boxes are stored in outdoor core racks for future reference. Individually bagged samples are placed in security-sealed rice bags along with the sample list for delivery to the assay laboratory.

One (1) blank and one (1) certified reference material ("CRM" or "standard") are inserted for every twenty (20) samples. The laboratory is also asked to assay one (1) pulp duplicate for every twenty (20) samples.

For every 100 samples sent to the laboratory, the numbers ending in the following digits represent QA/QC samples:

- 15, 35, 55, 75, or 95 = standard;
- 17, 37, 57, 77, or 97 = pulp duplicate of preceding sample;
- 20, 40, 60, 80, or 00 = blank.

11.2 Laboratory Accreditation and Certification

The International Organization for Standardization ("ISO") and the International Electrotechnical Commission ("IEC") form the specialized system for worldwide standardization. ISO/IEC 17025 General Requirements for the Competence of Testing and Calibration Laboratories sets out the criteria for laboratories wishing to demonstrate that they are technically competent, operating an effective quality system, and able to generate technically valid calibration and test results. The standard forms the basis for the accreditation of competence of laboratories by accreditation bodies.

Samples from the 2013 to 2018 drill programs were sent to Techni-Lab S.G.B. Abitibi Inc. ("Techni-Lab") in Sainte-Germaine-Boulé, Quebec, for preparation and analysis. Techni-Lab received ISO/IEC 17025 accreditation through the Standards Council of Canada ("SCC"). Techni-Lab is a commercial laboratory independent of the issuer and has no interest in the Project.

11.3 Laboratory Preparation and Assays

Samples were analyzed for gold using fire assay with atomic absorption spectroscopy ("AAS") finish. The nominal sample weight was 50 g.

The methodology is described as follows:

- Samples are sorted, bar-coded and logged into the Techni-Lab LIMS program before being placed in the sample drying room.
- Samples are crushed in their entirety to 85% passing 8 mesh (2.4 mm) using either an oscillating jaw crusher or a roll crusher. A 250 to 300 g fraction derived from the crushing process is pulverized using a ring mill to 90% passing 150 mesh (106 µm).
- Assay results are provided in Excel spreadsheets, and the official certificate (signed and sealed) is provided as a PDF file.
- The pulverized pulp is placed in kraft sample bags, and the un-pulverized portions returned to their original sample bags
- The remainder of the crushed samples (the rejects) and the pulps are returned to the client and stored at the Beattie mine facility.

Samples with grades over 5.0 g/t Au are re-assayed with a gravimetric finish. If the assay result from the gravimetric finish exceeds 10 g/t Au, then the sample is re-assayed by the metallic sieve method.

11.4 Quality Assurance and Quality Control

The quality assurance and quality control ("QA/QC") program for drill core includes the insertion of blanks, standards and duplicates in the sample stream of core samples. About 15% of the samples were control samples in the sampling and assaying process. One (1) standard, one (1) blank sample of barren rock and one (1) pulp duplicate were added to each group of 20 samples as an analytical check for the laboratory batches.

Geologists were responsible for the QA/QC program and database compilation. Upon receiving the analytical results, the geologists extracted the results for blanks and standards to compare against the expected values. If QA/QC acceptability was achieved for the analytical batch, the data were entered into the project's database; if not, the laboratory was contacted to review and address the issue, including retesting the batch if required.

The discussion below details the results of the blanks, standards and pulp duplicates used in the issuer's QA/QC program.

11.4.1 Certified Reference Materials (Standards)

Accuracy is monitored by inserting CRMs at a ratio of one (1) for every 20 samples (1:20). The standards were supplied by CDN Resource Laboratories Ltd. of Langley, British Colombia. A QC failure is defined as when the assay result for a standard falls outside three standard deviations ("3SD"). Gross outliers are excluded from the standard deviation calculation.

Fourteen (14) different standards were used between 2013 and 2018. Of the 310 CRM samples, eight (8) returned results outside 3SD (Table 11-1).

Table 11-1 – Results of standards used between 2013 to 2018

CRM	No. Of Assays	CRM value (Au g/t)	Average (Au g/t)	Accuracy (Au g/t)	Precision (%)	Outliers	Gross Outliers	Percent passing QC
CDN-GS-1D	3	1.05	1.04			0	0	100
CDN-GS-2F	1	2.16	2.37			0	0	100
CDN-GS-1J	35	0.95	0.97	2.4	7.2	1	0	97
CDN-GS-1P5F	34	1.40	1.44	3	6.0	0	1	97
CDN-PGMS-23	33	0.49	0.50	1.8	6.7	0	0	100
CDN-GS-2J	3	2.36	2.38			0	0	100
CDN-GS-1K	27	0.87	0.84	-2.7	9.6	0	1	96
CDN-GS-2K	35	1.97	2.04	4	7	1	0	97
CDN-GS-2L	33	2.34	2.45	4.7	4.8	0	0	100
CDN-GS-P3B	3	0.41	0.41			0	0	100
CDN-GS-P3C	35	0.26	0.26	-1.2	6.4	0	1	97
CDN-GS-P7E	35	0.77	0.78	1.6	6.4	1	0	97
CDN-GS-P7H	32	0.80	0.79	-0.6	4.8	1	0	97
CDN-GS-P8	1	0.78	0.82			0	0	100

Of the eight (8) fails, three (3) were identified as gross outliers. The issuer took action to explain the cause of the abnormal values. The results for two had been an inversion between two standards (CDN-GS-1P5F and CDN-GS-1K). The third seemed to be a problem at the laboratory (a slight underestimation of the grade) because the other standards in the batch had passed the QC. As the possible under-estimation was minor, the results were considered valid.

The overall success rate was 98%. Outliers did not generally show persistent analytical bias (either below or above the 3SD limit). They were close to the 3SD threshold and appeared to be isolated errors, as other standards and blanks processed from the same batches had passed. Consequently, no batch re-runs were performed. Figure 11-1 shows an example of a control chart for the standard CDN-GS-P7H assayed by Techni-Lab. A similar control chart was prepared for each CRM to visualize the analytical concentration value over time.

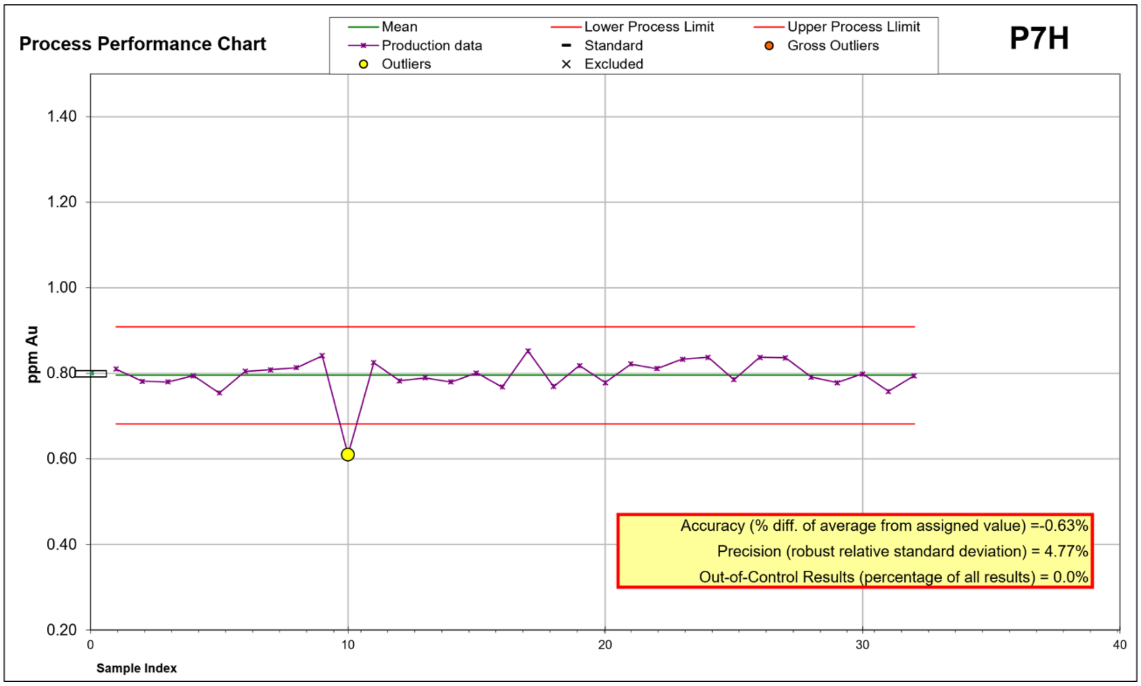


Figure 11-1 – Control chart for standard CDN-GS-P7H assayed by Techni-Lab (Innovexplo)

The overall results exhibit a slight positive bias in terms of accuracy, with an average of +1.5%, and a precision of around 6.1% for standards.

Both parameters meet standard industry criteria.

11.4.2 Blank Samples

Contamination is monitored by the routine insertion of a barren sample (blank), which goes through the same sample preparation and analytical procedures as the core samples.

A total of 313 blanks were inserted in the sample batches from 2013 to 2018. The blank material consisted of crushed marble. A general guideline for success during a contamination QC program is a rate of 90% of blank assay results not exceeding the acceptance limits of three times the detection limit (“3DL”). The detection limit was 0.01 g/t Au.

One (1) sample did not pass the quality control procedure, representing a success rate of 99.6% (Figure 11-2)

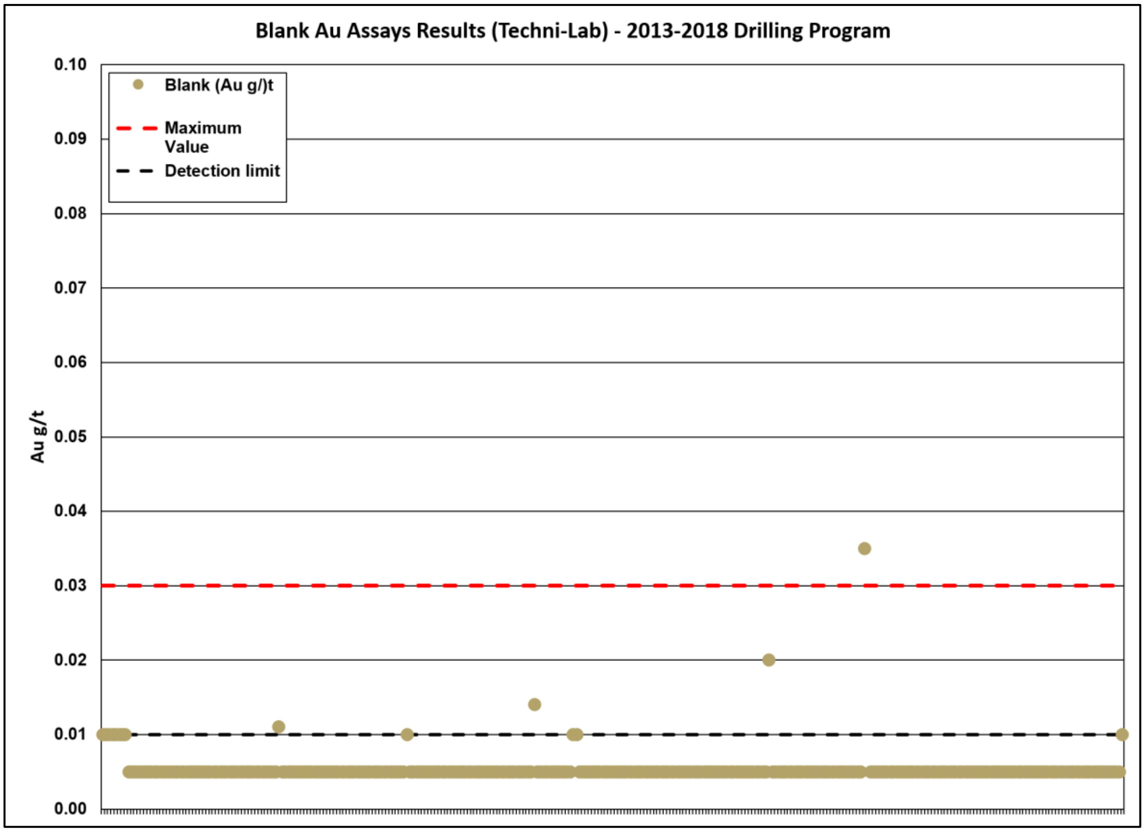


Figure 11-2 – Time series plot for blank samples assayed by Techni-Lab between 2013 to 2018 (Innovexplo)

11.4.3 Pulp Duplicates

The precision of the pulp duplicates can be used to determine the incremental loss of precision for the pulp pulverizing stage of the process, thereby establishing whether a given pulp size taken after pulverization is adequate to ensure representative fusing and analysis.

A total of 306 pulp duplicates was assayed. The difference between the original and duplicate analyses is presented in the Figure 11-3 scatter plot. Results show a good precision with R_2=0.94. Results also show a good accuracy monitored by the linear regression line (between the 10% tolerance limit).

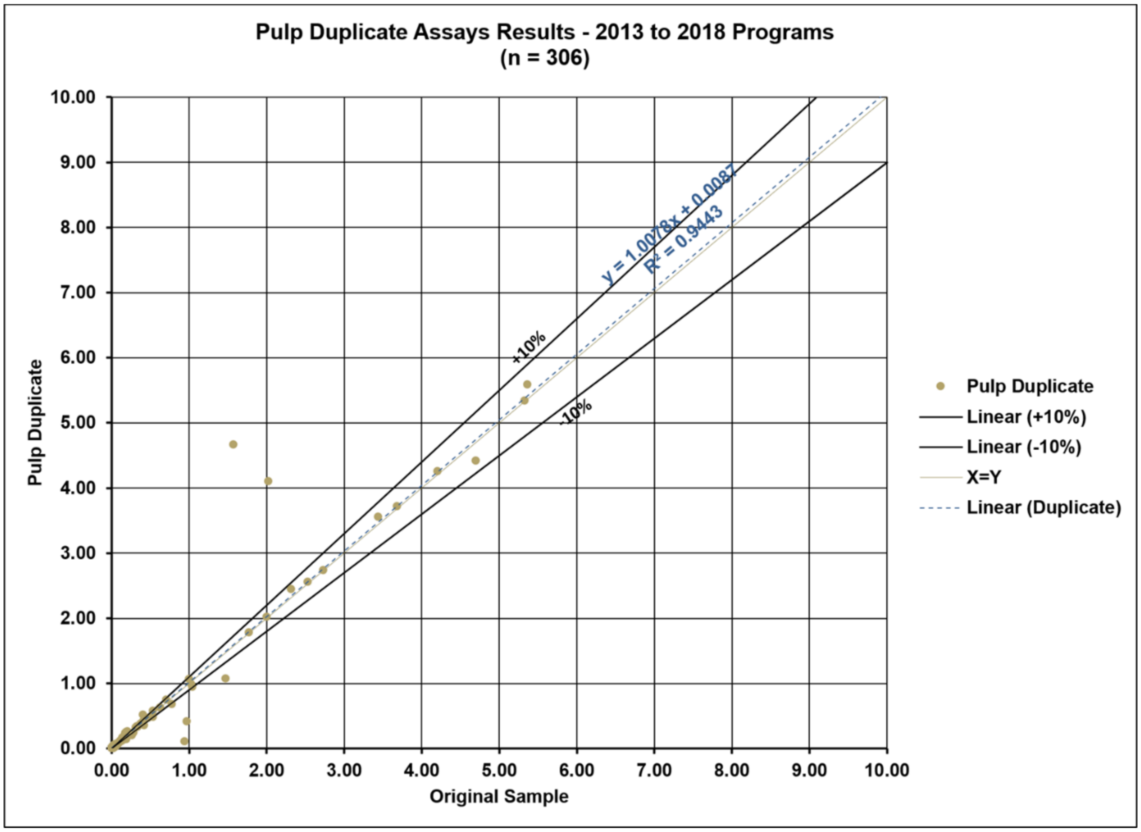


Figure 11-3 – Linear graph comparing original and pulp duplicate samples analyzed by Techni-Lab between 2013 to 2018

11.5 Conclusion

The authors are of the opinion that the sample preparation, security, analysis and QA/QC protocols performed between 2013 to 2018 followed generally accepted industry standards and that the data is valid and of sufficient quality for mineral resource estimation.

12. DATA VERIFICATION

This item covers the verification of all the new data supplied by the issuer that were added to the previously validated diamond drill hole database used for the 2014 MRE (Poirier et al., 2014). The close-out date of the 2014 MRE database was February 28, 2013. The close-out date for the current MRE database is October 20, 2021.

Data verification included visits to the Project as well as an independent review of the data for selected drill holes (surveyor certificates, assay certificates, QA/QC program and results, downhole surveys, lithologies, alteration and structures).

12.1 Site Visits

Marina Iund (P.Geo.) visited the Project on October 20, 2021, and Carl Pelletier (P.Geo.) on November 16, 2011, and February 7, 2012. Onsite data verification included a general visual inspection of the Project and the core storage facilities, a check of drill collar coordinates, and a review of selected mineralized core intervals, the QA/QC program and the log descriptions of lithologies, alteration and mineralization.

12.2 Core Review

The core boxes are stored in core racks. The authors found the core boxes to be in good order and properly labelled, and the sample tags were present. The wooden blocks at the beginning and end of each drill run were still in the boxes, matching the indicated footage on each box. The authors validated the sample numbers and confirmed the presence of mineralization in the reference half-core samples (Figure 12-1).

A) Core racks; B and C) Proper labelling of the core racks and drill core boxes; D) Sample tag stapled in core box; E) Mineralization from hole BD10-117

Figure 12-1 – Photographs taken during the drill core review

12.3 Database

Fifty-seven (57) new diamond drill holes were added to the previously compiled and verified master database (Poirier et al., 2014):

- 24 from Beattie
- 27 from Donchester
- 6 from Central Duparquet

Two holes drilled on the Beattie Project in 2014 and 2015 (BD-14-01 and BD-15-01) had undergone later additional sampling of the mineralized zones as seen by the inspection of the drill core, however the assay results were not reported in the drill logs or available, those holes were not included in the database.

The updated master database (the “First Mining Database”) contains 904 holes totalling 270,119 m and 173,831 sampled intervals.

12.3.1 Drill Hole Locations

Collar position coordinates and azimuths are presented in the database using the UTM system (NAD 83, Zone 17).

The drill hole collars from the 2013 to 2018 diamond drilling programs were surveyed by Patrick Descarreaux Arpenteur-Géomètre Inc. of La Sarre using a Differential GPS with an established base station.

The coordinates of six (6) surface holes were confirmed by the author using a handheld GPS (Figure 12-2 and Table 12-1), then compared to the database. All results had acceptable precision.

The collar locations in the First Mining Database are considered adequate and reliable.

A) D13-27 collar; B) B17-01 collar

Figure 12-2 – Examples of onsite verification of collar locations

Table 12-1 – Original collar survey data compared to InnovExplo's checks

Hole ID	Original coordinates		Checked coordinates		Difference (m)	
	Easting	Northing	Easting	Northing	Easting	Northing
BD-16-01	630841.3	5374343.9	630840	5374345	1.3	-1.1
BD-13-38	630258.1	5374228.8	630258	5374233	0.1	-4.2
BD-17-01	631409.9	5374281.6	631409	5374291	0.9	-9.4
BD-18-01	631417.6	5374289.2	631417	5374290	0.6	-0.8
D-13-20	631627.2	5374495.7	631626	5374497	1.2	-1.3
D-13-27	632098.3	5374409.9	632090	5374410	8.3	-0.1

12.3.2 Downhole Survey

Downhole surveys (Acid, Gyro, Pajari, Deviflex, Flexit and Reflex) were conducted on the majority of surface holes. The downhole survey information was verified for 5% of the holes used in the 2022 MRE. The holes were selected based on their representativeness, both in terms of the drilling program they were part of (more focus on new drilling programs) and their geographical position with respect to the interpreted mineralized zones.

Minor errors of the type normally encountered in a project database were identified and corrected.

12.3.3 Assays

The author had access to the assay certificates. The assays in the database were compared to the original certificates provided by the laboratory. The verified holes represent 5% of the First Mining Database. The holes were selected based on their representativeness, both in terms of the drilling program they were part of (more focus on new drilling programs) and their geographical position with respect to the interpreted mineralized zones.

Minor errors of the type normally encountered in a project database were identified and corrected.

12.3.4 Channel Sample Data

A total of 2,371 samples from 892 channels (for a total length of 1,827 m) had already been entered and validated in the master database (Poirier et al., 2014). As no new channel samples have been provided by the issuer, no further verification of the channel sample data was deemed necessary.

The results of the 2013 channel sampling program carried out by Clifton Star were not included in the MRE 2014, nor could they be included in the current MRE as the results were not available.

12.4 Conclusion

The author believes that the data verification process demonstrates the validity of the data and the protocols for the Project. The author considers the database for the Project to be valid and of sufficient quality to be used for the mineral resource estimate herein.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Metallurgical Testing

13.1.1 Introduction

Bench-scale and pilot plant metallurgical testwork programs have been carried out for the Project. SGS carried out preliminary metallurgical testwork in 2012 to support a PEA at that time (reference #12 below). In 2013, SGS carried out further flotation, pressure oxidation, cyanidation, rheology and environmental testwork, including a pilot plant for a PFS. Outotec was also mandated in 2013 to carry out filtration testwork. The relevant reports from these test programs are listed below. Numbers inside square brackets refer to this list.

SGS Canada

1. An investigation into the recovery of gold from Duparquet Project Samples, Project 13054-001 Final Report, May 24, 2012
2. An investigation into the recovery of gold from Duparquet Project Samples, Project 13054-002 Report 1, September 27, 2012
3. An investigation into environmental characterization of minerals processing residues from the Duparquet Project, Project 13054-002 Report 2, November 29, 2012
4. An investigation into cleaner flotation for the recovery of gold from Duparquet Project Samples, Project 13054-003 Final Report, February 5, 2013
5. An investigation into the grindability characteristics of a pilot plant feed sample from the Duparquet deposit, Project 13054-004 Grindability Report, May 21, 2013
6. A flotation pilot plant investigation into the recovery of gold from Duparquet samples, Project 13054-004 Report 1, June 20, 2013
7. A pilot plant investigation into pressure oxidation followed by hot curing of Duparquet flotation concentrate prepared for Clifton Star Resources Inc. Project 13054-004 Report #2, August 26, 2013
8. An investigation into the rheometallurgical response of Duparquet metallurgical samples, Project 13054-001 – Draft - Report #3, August 27, 2013
9. An investigation into the design of a carbon adsorption circuit and downstream testing for the Duparquet deposit, Project 13054-004 Report #4, September 12, 2013
10. An investigation into the grindability characteristics of a pilot plant feed sample from the Duparquet deposit, Project 13054-004 Grindability Report – Rev 1, September 20, 2013

Outotec

1. Filtration test report, Project 108096T 1, August 15, 2013

Other Reports

1. Williamson et al., Preliminary Economic Assessment, 2013

The sections that follow review the data presented in each of these reports and how the data have been used to establish the process flowsheets, design criteria, and process plant design for the PFS published in 2014.

The samples used for the various testwork programs were selected under the supervision of InnovExplo and Clifton Star as described in Section 11.7.

13.1.2 Comminution

13.1.2.1 Bench Tests

The testwork was conducted on six (6) mineralized material samples and two (2) in-situ tailing samples [1]. The gold grade of the mineralized material samples ranged from 1.3 to 3.5 g/t Au. The sulphide content ranged from 0.5% to 2.0% S.

The Bond Work Index of the mineralized material samples varied from 17.2 to 20.2 kWh/t, classifying them as hard to very hard mineralized material.

13.1.2.2 Pilot Plant Tests

SGS Project Reference No. 13054-004 – Grindability Report [5] includes a series of grindability tests that have been conducted on a pilot plant (PP Feed) sample. The sample was full HQ core, consisting of about 12 tons of material, assumed to be representative of the Duparquet Project. The results are summarized in Table 13-1.

Table 13-1 – Grindability Test of Pilot Plant Feed

Sample Name	Relative Density	JK Parameters			Locked-cycle		HPGR	Batch	CWI	RWI	BWI (kWh/t)		AI
		Axb[1]	Axb[2]	ta	kWh/t	N/mm^2	ts/hm^3	kWh/t	kWh/t	kWh/t	Feed	HPGR Prod	(g)
PP Feed	2.73	27.0	29.0	0.20	2.82	3.42	233	1.97	7.5	19.1	18.5	16.6	0.547
Bulk PP Feed*	-	-	-	-	-	-	-	-	-	-	17.6	-	-
DON-11-147M	2.74	-	34.4	-	-	-	-	-	-	18.4	18.8	-	0.472
BD-11-333M	2.68	-	25.0	-	-	-	-	-	-	18.2	18.5	-	0.668
BD-11-336M	2.53	-	25.9	-	-	-	-	-	-	18.5	20.3	-	0.618
CD-11-01MM	2.88	-	31.0	-	-	-	-	-	-	17.3	18.4	-	0.836

1. A x b from DWT
2. A x b from SMC

Mill feed sample collected after the bulk sample crushing

The characterization of the composite sample included the following grindability tests:

- Bond low-energy impact test
- JKTech drop-weight test
- SMC test
- Bond rod mill grindability test (“RWI”)
- Bond ball mill grindability test (“BWI”)
- Bond abrasion test
- HPGR test

The Pilot Plant Feed sample was characterized as very hard with respect to both resistance to impact (Axb) and abrasion breakage (ta), as well as in terms of RWI. The sample was characterized as hard with respect to the BWI and HPGR tests. The sample was also found to be very abrasive. All the results were in line with the six (6) previous samples, tested as part of SGS Project 13054-002 [2].

SGS Project Reference No. 13054-004 –Grindability Report – REV1 [4] includes the results of a single JKSimMet simulation that was conducted, using aforementioned grindability test results on the PP Feed sample. The results of the simulations are presented in the Table 13-2.

Table 13-2 – JKSimMet Simulation Results

SAG Mill Circuit	
Sample Name	PP Feed
Total SAG #	1
Mill Size (Inside Shell Diam. X EGL)	30.0' x 11.0'
Mill Size (Inside Liner Diam.. x EGL)	8.94 m x 3.35 m
Total Feed Rate, t/h	452.9
Total Feed Rate, tpd	10,000
Circuit Availability, %	92
Fresh Feed Size, mm, F80	157
Fresh Feed Size, mm,%-12.7	21.3
Fresh Feed Axb	27.0
Ball Charge,%Vol	12
Mill Speed, % of Cr.	75
SAG Discharge% Solids, w/w	65
Grate Size, mm	64
Class. Slots Size, mm	12.7
SAG Recycle, t/h	99
SAG Recycle, %	22
Pebble Crusher, Y/N	Y
T80, µm	2,125
Total SAG Mill Power Requirements (Motor input), kW	4,551
Total SAG Mill Power Requirements (Motor input), kWh/t	10.0

Ball Mill Circuit	
Sample Name	PP Feed
Total BM #	1
Mill Size (Inside Shell Diam. X EGL)	20.0' x 30.0'
Mill Size (Inside Liner Diam. x EGL)	5.94 m x 8.99 m
Total Feed Rate, t/h	452.9
Total Feed Rate, tpd	10,000
F80, µm	2,125
Make-up Ball Size Diam., mm	57
Ball Charge,%Vol	32
Mill Speed, % of Cr.	75
BM Discharge% Solids, w/w	75
Bench-Scale Work Index (kWh/t), RWI	19.1
Bench-Scale Work Index (kWh/t), BWI	18.5
Recycle, t/h	1,132
Recycle, %	250
P80, µm	100
Total Ball Mill Power Requirements (Motor input), kW	5,674
Total Ball Mill Power Requirements (Motor input), kWh/t	12.5

Overall Circuit	
Sample Name	PP Feed
Total SAG #	1
Total Feed Rate, tpd	10,000
F80, mm	157
P80, µm	100
Total Mill Power Requirements (Motor input), MW	10.2
Total Mill Power Requirements (Motor input), HP	13,706
Total Mill Power Requirements (Motor input), kWh/t	22.6
Operating Work Index (Wio), kWh/t	22.0
SAG Motor, MW	5.4
SAG Motor, HP	7,200
SAG Motor, % Utilization	84
BM Motor, MW	6.4
BM Motor, HP	8,600
BM Motor, % Utilization	89
Total Installed Power,	12.0
Total Installed Power,	15,800

The simulations were performed with a SAG mill charge of 25%, 12.7 mm classification screen aperture and a maximum of 25% recycle rate. The SAG mill speed was assumed at 75% of the critical speed.

In the JKSimMet simulation, it was found that a 30.0’ x 11.0’ (nominal diameter x EGL) SAG mill, operated with a 12% ball charge would be required to grind 453 t/h crushed mineralized material with F_{80} of 157 mm to a product with P_{80} of 2 mm. The SAG mill and ball mill specific power requirements were 10.0 and 12.5 kWh/t, respectively for a total power consumption of 22.6 kWh/t. It was also found that 5.7 MW would be required for the ball mill circuit to achieve a final product size of 100 µm. One 20.0’ x 30.0’ ball mill, with internal dimensions of 5.94 x 8.99 m and assumed 0° cone angle (square mill equivalent), operating with a ball charge of about 32% was selected. The SAG and ball mill installed power should be 4,551 kW and 5,674 kW, respectively. The SAG mill motor was selected to allow a ball charge increase up to 15% as well as an increase to 78% of critical speed.

13.1.3 Gravity Separation

The preliminary gravity separation testwork [1] on ground mineralized material samples indicated low gold recovery ranging from 3.7% to 14.9% and averaging 8.6%. Hence, gravity separation was not pursued.

13.1.4 Flotation

13.1.4.1 Bench Tests

Benchscale flotation tests were conducted on the six mineralized material samples and in situ tailings samples evaluated for BWI (13.1.2.1). As shown in Table 13-3 the recovery of gold to concentrate by flotation [1] was greater than 90% for most samples. The Central Duparquet Main sample is the exception with 84.6% gold recovery, increasing to 87.9% with finer grinding. Gold recovery by pressure oxidation and cyanidation treatment of the flotation concentrate was also investigated and was consistently high.

Table 13-3 – Summary of Overall Results for the mineralized material Sample

Sample	Flotation Concentrate				POX Time min	Concentrate POX-Cyanidation							Tailing Cyanidation							Overall Au Recovery, %			Comb Tail Au, g/t	Head (calc) Au, g/t	Head (direct) Au, g/t
	Test No.	Wt %	Assay Au, g/t	Rec'y %, Au		Test No.	Add'n, kg/t of mineralized material		Cons., kg/t of mineralized material		Extr'n Au	Residue Au, g/t	Test No.	Add'n, kg/t of mineralized material		Cons., kg/t of mineralized material		Extr'n Au	Residue Au, g/t	Conc	Tail	Comb Rec'y			
							NaCN	CaO	NaCN	CaO				NaCN	CaO	NaCN	CaO								
A Zone	F-9	15.2	18.0	94.6	60	CIL2	0.14	0.83	0.03	0.83	79.8	3.72	CN-7	0.76	0.42	0.09	0.42	26.7	0.22	75.5	1.4	76.9	0.75	2.89	3.54
	F-9	15.2	18.0	94.6	90	CIL12	0.19	6.33	0.14	6.33	98.5	0.25	CN-7	0.76	0.42	0.09	0.42	26.7	0.22	93.2	1.4	94.6	0.22		
South Zone	F-10	12.7	9.76	91.8	60	CIL3	0.12	0.56	0.03	0.48	97.6	0.23	CN-8	0.71	0.37	0.10	0.37	56.3	0.06	89.6	4.6	94.2	0.08	1.35	1.55
	F-10	12.7	9.76	91.8	90	CIL13	0.13	0.83	0.02	0.83	98.9	0.11	CN-8	0.71	0.37	0.10	0.37	56.3	0.06	90.8	4.6	95.4	0.07		
RW Zone	F-12	19.9	15.8	94.4	60	CIL4	0.19	0.42	0.05	0.41	76.3	3.95	CN-10	0.57	0.69	0.02	0.62	38.6	0.13	72.0	2.2	74.2	0.89	3.33	3.32
	F-12	19.9	15.8	94.4	90	CIL15	0.23	8.88	0.18	8.88	97.0	0.46	CN-10	0.57	0.69	0.02	0.62	38.6	0.13	91.6	2.2	93.7	0.20		
Donch-ester	F-11	18.2	10.1	91.3	60	CIL5	0.21	4.36	0.14	4.36	98.1	0.17	CN-9	0.69	0.55	0.18	0.55	43.3	0.11	89.6	3.8	93.3	0.12	2.01	2.25
North	F-11	18.2	10.1	91.3	90	CIL14	0.15	5.68	0.04	5.68	97.4	0.25	CN-9	0.69	0.55	0.18	0.55	43.3	0.11	88.9	3.8	92.7	0.14		
Donch-ester	F-13	18.5	7.22	92.4	60	CIL6	0.26	5.28	0.08	5.28	94.8	0.16	CN-11	0.75	0.44	0.24	0.44	26.2	0.09	87.6	2.0	89.6	0.10	1.44	1.29
South	F-13	18.5	7.22	92.4	90	CIL16	0.17	4.40	0.02	4.40	97.1	0.20	CN-11	0.75	0.44	0.24	0.44	26.2	0.09	89.7	2.0	91.7	0.11		
Central	F-14	22.3	8.85	84.6	60	CIL7	0.28	1.98	0.11	1.98	94.9	0.43	CN-12	0.55	0.51	0.04	0.48	50.0	0.28	80.3	7.7	88.0	0.31	2.33	2.33
Dupar-quet	F-14	22.3	8.85	84.6	90	CIL17	0.19	1.83	0.02	1.82	95.8	0.36	CN-12	0.55	0.51	0.04	0.48	50.0	0.28	81.0	7.7	88.7	0.30		
Main	F-17	23.9	9.27	87.9	60	CIL18	0.37	2.53	0.11	2.53	98.2	0.16	CN-12*	0.54	0.50	0.04	0.47	50.0	0.28	86.3	6.1	92.4	0.25		

*CN12 from F-14 (not F-17)

13.1.4.2 Locked-Cycle Tests

SGS Project Reference No. 13054-003 – Final Report [3] summarizes the results of the flotation testwork, conducted to investigate the recovery of gold from six (6) mineralized material samples. This program also included detailed concentrate analysis and cyanidation tests on the flotation tailings samples.

Cleaner flotation tests were conducted to investigate the recovery of the gold in a saleable sulphide concentrate. The rougher concentrate with P_{80} of ~100 µm was cleaned twice and a scavenger stage was applied following the first cleaner tailings. The results of the locked cycle tests conducted on each sample are presented in Table 13-4.

Table 13-4 – Flotation Locked-Cycle Test Results

Test No. Sample	Grind P80, µm	Product	Wt%	Assays		% Distribution	
				Au g/t	$S^{=}$%	Au g/t	$S^{=}$%
LCT1	115	2nd Cleaner Conc.	3.8	83.6	35.1	88.5	88.5
A Zone		1st Cleaner Scav. Tail	20.3	0.99	0.4	5.6	5.4
		Rougher Tail	75.9	0.27	0.12	5.9	6.1
		Head (Calculated)	100	3.55	1.49	100	100
LCT2	113	2nd Cleaner Conc.	1.9	67.5	27.5	84.5	88.6
South		1st Cleaner Scav. Tail	23	0.49	0.12	7.3	4.6
		Rougher Tail	75.1	0.17	0.05	8.3	6.8
		Head (Calculated)	100	1.54	0.6	100	100
LCT3	109	2nd Cleaner Conc.	6.3	50.2	29.7	87.3	92.2
RW Zone		1st Cleaner Scav. Tail	22.7	0.97	0.34	6.1	3.8
		Rougher Tail	71.1	0.33	0.11	6.6	4.0
		Head (Calculated)	100	3.6	2.02	100	100
LCT4	108	2nd Cleaner Conc.	3.2	56.7	35.9	83.1	86.6
Donchester N		1st Cleaner Scav. Tail	16.4	0.91	0.5	6.9	6.2
		Rougher Tail	80.5	0.27	0.12	10	7.1
		Head (Calculated)	100	1.43	0.87	100	100
LCT5		2nd Cleaner Conc.	2.9	39.5	32	82.9	81.2
Donchester S	108	1st Cleaner Scav. Tail	16.5	0.65	0.5	7.8	7.2
		Rougher Tail	80.6	0.16	0.16	9.4	11.5

Test No. Sample	Grind P80, µm	Product	Wt%	Assays		% Distribution	
				Au g/t	S⁼%	Au g/t	S⁼%
		Head (Calculated)	100	1.38	1.14	100	100
LCT6	100	2nd Cleaner Conc.	4.8	39	20.9	75.5	83.5
Central		1st Cleaner Scav. Tail	25.1	1.02	0.37	10.2	7.7
Duparquet		Rougher Tail	70.7	0.51	0.15	14.2	8.9
Main		Head (Calculated)	100	2.49	1.21	100.2	100

The gold recovery ranged from 75.5% (Central Duparquet Main sample) to 88.5% (A Zone sample). The cleaner concentrate gold grade ranged from 39.0 to 83.6 g/t Au and the sulphur grade ranged from 20.9% to 35.9% S.

13.1.4.3 Pilot Plant Tests

SGS Project Reference No. 13054-004 – Report 1 [6] summarizes the results of the flotation tests conducted on a pilot plant (PP Feed) sample. The head sample was analyzed at 1.84 g/t Au, 1.16% S, 0.055% As and 7.61% CO_3. The pilot plant was operated to generate bulk sulphide flotation concentrate containing 15-18% S for a subsequent pressure oxidation pilot plant to assess gold recovery and to generate 60-80 kg of a higher-grade flotation concentrate assaying over 40 g/t Au for direct sale market evaluation.

In the flotation pilot plant, the mineralized material was ground to a P_{80} of 100 µm and a rougher concentrate was recovered with addition of PAX and R208 collectors. The rougher concentrate was reground in a ball mill, operating in closed circuit with a cyclone.

Pilot plant tests PP-01 to PP-06 were conducted with one cleaning stage to generate the pressure oxidation pilot plant feed. Pilot plant tests PP-07 to PP-09 were conducted with two cleaning stages to generate the high-grade concentrate. Tests PP-05 and PP-09 were operated over more than 8 hours under relatively stable conditions. The results of these tests are summarized in Table 13-5.

With one cleaning stage (PP-05) the recovery of gold was 91.7% in a concentrate containing 26.8 g/t Au and 16.1% S. The results of PP-09 indicated that a concentrate with 47.8 g/t Au could be produced at 86.5% gold recovery. Flotation tailings were leached with cyanide to recover the gold remaining in this material. The recovery of gold from the PP-05 tailings and the PP-09 tailing were 45.1% and 40.2%, respectively. The overall gold recovery by flotation and cyanidation of the flotation tailings is shown in Table 13-6.

Table 13-5 – Pilot Plant Flotation Test Results at Steady State

Test	P_{80} µm	Product	Wt%	Assays, g/t, %		Distribution, %	
				Au	S	Au	S
PP5	99	PP Feed	100	1.81	1.09	100	100
		Rougher Feed	113.9	1.72	1.06	108.4	110.5
		Rougher Conc.	20.1	9.01	5.51	100.1	101.8
		Rougher Tail	93.8	0.16	0.1	8.3	8.6
		1st Cleaner Conc.	6.2	26.8	16.1	91.7	91.4
		1st Cleaner Tail	13.9	1.09	0.82	8.4	10.5
PP9	100	PP Feed	100	1.81	1.12	100	100
		Rougher Feed	108.1	1.91	1.22	113	117.1
		Rougher Conc.	12.4	17.1	11.1	99.5	104.3
		Rougher Tail	95.7	0.25	0.15	13.5	12.8
		1st Cleaner Feed	20.6	16.6	10.9	106.4	112.5
		1st Cleaner Conc.	12.5	39	24.6	93.5	95.4
		1st Cleaner Tail	8.1	3.23	2.64	13	17.1
		2nd Cleaner Conc.	4.3	47.8	29.8	86.5	87.2
		2nd Cleaner Tail	8.2	11.9	8.71	7	8.2

Table 13-6 – Overall Gold Recovery by Flotation and Cyanidation of the Flotation Tails

Sample	Concentrate Grade		Overall Au Recovery, %		
	Au, g/t	S, %	Flot. Conc.	Tail CN	Total
PP-05 (Conc for POX feed)	26.8	16.1	91.7	3.7	95.4
PP-09 (Conc for direct sale)	47.7	29.8	86.5	5.4	91.9

13.1.5 Cyanidation on Flotation Tailings

13.1.5.1 Bench Tests

Samples of the benchscale flotation tailings [1] presented in section 13.1.4.1 were leached under conventional cyanidation conditions. Standard bottle roll tests were conducted at 40% solids and pH 10.5 with 0.5 g/L NaCN for 48 hours. These tests gave poor gold extractions varying from 26.2% to 56.3%, confirming the refractory nature of the mineralized material.

13.1.5.2 Locked-Cycle Tests

The rougher tailings and cleaner scavenger tailings from each flotation locked cycle test (see section 13.1.4.2) were leached separately to investigate the gold extraction.

Table 13-7 indicates the leaching efficiency of cyanidation of the flotation tailings. Depending on the sample, between 3.9% and 11.5% additional gold was leached by cyanidation of flotation tailings.

Table 13-7 – Cyanidation of Flotation Tailings

Test No. Sample	Product	Wt%	Assays Au, g/t	Distr'n Au, %	Extr'n Au, %	Overall Au Rec. %
LCT1	1st Cleaner Scav. Tail	20.3	0.99	5.6	41.5	2.3
A Zone	Rougher Tail	75.9	0.27	5.9	27.3	1.6
South	1st Cleaner Scav. Tail	23	0.49	7.3	51.1	3.7
	Rougher Tail	75.1	0.17	8.3	45.5	3.8
RW Zone	1st Cleaner Scav. Tail	22.7	0.97	6.1	39.9	2.4
	Rougher Tail	71.1	0.33	6.6	28.9	1.9
Donchester N	1st Cleaner Scav. Tail	16.4	0.91	6.9	51.2	3.5
	Rougher Tail	80.5	0.27	10	40.3	0.4
Donchester S	1st Cleaner Scav. Tail	16.5	0.65	7.8	35.4	2.8
	Rougher Tail	80.6	0.16	9.4	20.4	1.9
Central	1st Cleaner Scav. Tail	25.1	1.02	10.2	51.2	5.2
Duparquet	Rougher Tail	70.7	0.51	14.2	44.3	6.3

The overall gold recovery ranged from 87.0% (Central Duparquet Main sample) to 92.4% (A Zone sample). The overall reagent consumptions in the cyanidation tests are given in Table 13-8.

Table 13-8 – Overall Reagent Consumptions for Tailing Cyanida

Sample	Rougher Tailing				Cleaner Scavenger Tailing				Combined Tailing			
	Reagent Add'n, kg/t		Reagent Cons, kg/t		Reagent Add'n, kg/t		Reagent Cons, kg/t		Reagent Add'n, kg/t		Reagent Cons, kg/t	
	NaCN	CaO	NaCN	CaO	NaCN	CaO	NaCN	CaO	NaCN	CaO	NaCN	CaO
A Zone	0.58	0.41	0.06	0.41	0.38	0.19	0.1	0.19	0.96	0.60	0.16	0.60
South	0.56	0.34	0.05	0.34	0.52	0.22	0.1	0.22	1.08	0.56	0.14	0.56
RW Zone	0.61	0.53	0.18	0.53	0.62	0.29	0.28	0.29	1.23	0.81	0.47	0.81
Doncheste	0.67	0.43	0.11	0.43	0.34	0.17	0.12	0.17	1.01	0.61	0.23	0.61
Doncheste	0.76	0.56	0.21	0.56	0.35	0.19	0.14	0.19	1.11	0.76	0.35	0.76
Duparquet	0.74	0.34	0.16	0.33	0.6	0.21	0.13	0.21	1.34	0.55	0.29	0.54

The cyanide (NaCN) consumption for leaching the flotation tailings varied between 0.14 kg/t and 0.47 kg/t. For the same tests, the lime (CaO) consumption varied between 0.54 g/t and 0.81 kg/t.

13.1.6 Pressure Oxidation (POX) and Cyanidation

13.1.6.1 Bench Tests on Tailings

Testwork was also performed on two samples of existing tailings on the Project. The overall results for the flotation-POX/CIL flowsheet for both tailing samples are summarized in Table 13-9. The overall recovery of gold was 83.5% for sample T-1 and 93.3% for sample T-2.

Table 13-9 – Summary of Overall Results for the Tailing Samples

Sample	Flotation Concentrate				POX Time min	Conc POX-Cyanidation			Tailling Cyanidation			Overall Recovery, %			Comb Tail Au, g/t	Head (Calc) Au, g/t	Head (direct) Au, g/t
	Test No.	Wt %	Assay Au, g/t	Rec'y %, Au		Test No.	Extr'n Au	Residue Au, g/t	Test No.	Extr'n Au	Residue Au, g/t	Conc Au	Tail Au	Comb. Rec'y			
T-1 Tailing	F-18	24.7	3.59	72.1	90	CIL19	92.9	029	CN-19	59.1	0.22	67.0	16.5	83.5	0.24	1.23	1.18
T-2 Tailing	F-16	47.4	12.5	94.9	90	CIL1	97.3	0.30	CN-6	19.7	0.43	92.3	1.0	93.3	0.39	6.25	6.40

13.1.6.2 Bench Tests on mineralized material

This section refers to SGS Project Reference No. 13054-002 – Report 1 [2] which includes the metallurgical testwork that was conducted on six flotation concentrate samples produced in the previous 13054-001 test program [1].

The overall gold results [1] from flotation concentrate, pressure oxidation (POX) and CIL circuit as well as flotation tail cyanidation for the mineralized material are summarized in Table 13-3. The overall recovery of gold was ranging from 91.9% to 95.4%.

The first objective of this program was to attempt to reduce costs of the pressure oxidation (POX) and carbon-in-leach (CIL) process by optimizing conditions and reducing reagent requirements. Previous investigation showed high lime consumption in CIL which was attributed to the slow breakdown of basic iron sulphates produced during pressure oxidation. To address this problem, the POX products were kept at 95°C for 4 hours in what is known as a hot cure process allowing the precipitated basic iron sulphate to solubilise back into solution. By this approach, the lime consumption in the CIL circuit was reduced by up to 95%.

Acid additions in the pre-acidulation stage before pressure oxidation were reduced by approximately half (by 60 kg/t H_2SO_4) from the initial tests while maintaining the high degree of sulphide oxidation and high gold recoveries. In addition, the hot cure product solution was successfully used as the source of acid in the pre-acidulation step, eliminating the fresh acid requirement and the cost of neutralizing the POX product. Furthermore, the testwork showed similar gold recovery values in the range of 96-99% at a lower POX temperature of 210°C compared to 225°C applied in previous testwork. This would result in additional savings in pressure oxidation costs.

The second objective was to generate final products for environmental studies. This included the flotation tailings, detoxified CIL pulp and hot cure neutralization sludge with each sample to be evaluated separately as well as a combined tailing product which included all three tailing streams.

The results of the POX and CIL tests in terms of sulphide oxidation and gold recovery as well as reagent consumption are presented in Table 13-10.

Table 13-10 – Summary of Pressure Oxidation – CIL Testwork

Sample	Conc S^{2-}%	POX Test No.	Pre-acidulation H_2SO_4 Add'n, kg/t		S^{2-} oxid'n in POX %	Hot Cure h	Carbon-in-leach Results					
							Test No.	Reag. Cons., kg/t		% Extr'n Au	Residue Au. g/t	Head (calc) Au, g/t
			fresh	recycled				NaCN	CaO			
A Zone	6.54	POX21	121.2		99.3	0	CIL21	0.32	53.2	98.1	0.3	17.4
		POX27	107.8		99.4	4	CIL27	0.29	2.7	98.3	0.31	16.3
		POX33	58.8		99.4	4	CIL33	0.24	19.5	98.1	0.41	17.9
		POX36		58.8	99.4	4	CIL36	0.2	5.5	98.1	0.38	17.8
South Zone	4.15	POX22	99.1		99.0	0	CIL22	0.22	4.1	98.9	0.12	10.4
		POX28	94.8		99.3	4	CIL28	0.26	4.5	98.5	0.14	9.7
		POX38		47.4	99.3	4	CIL38	0.14	1.5	97.8	0.22	9.4
RW Zone	8.75	POX24	98.1		99.4	0	CIL24	0.46	53	97.9	0.3	15.3
		POX30	96.7		98.9	4	CIL30	0.26	3.6	97.1	0.44	14.3
		POX39		48.4	99.3	4	CIL39	0.14	2	97.8	0.38	14.5
Donchester N	6.38	POX23	167.1		98.8	0	CIL23	0.35	21.9	96.2	0.35	9.1
		POX29	157.4		99.1	4	CIL29	0.19	3.1	97.8	0.22	9.6
		POX37		100	98.8	4	CIL37	0.16	6.7	98.1	0.18	9.2
Donchester S	6.19	POX25	129.7		99.1	0	CIL25	0.29	18.3	97.9	0.16	7.5
		POX31	119.8		98.8	4	CIL31	0.26	4.4	97.5	0.18	6.6

Sample	Conc S^{2-}%	POX Test No.	Pre-acidulation H_2SO_4 Add'n, kg/t		S^{2-} oxid'n in POX %	Hot Cure h	Carbon-in-leach Results					
							Test No.	Reag. Cons., kg/t		% Extr'n Au	Residue Au. g/t	Head (calc) Au, g/t
			fresh	recycled				NaCN	CaO			
		POX40		59.6	99.1	4	CIL40	0.13	1.7	97.7	0.16	6.5
Duparquet Central	4.8	POX26	38.6		99.0	0	CIL26	0.22	8	94.5	0.43	7.5
		POX32	34.7		97.2	4	CIL32	0.28	3.1	95.5	0.38	8.5
		POX41		17.4	99.1	4	CIL41	0.1	1.4	96.4	0.34	8.8

Pressure oxidation optimization and cyanidation testwork

The pressure oxidation tests were carried out in a 2 L Parr autoclave [2]. For initial testing, the concentrates were pre-acidulated at pH 1.8 with sulphuric acid for 120 minutes as in the previous work. The POX temperature was lowered to 210°C from 225°C. The oxygen overpressure was 100 psi during the 90-minute residence time. As per established practice, the pulp density used for each sample was calculated based on its sulphide content. The autoclave discharge was filtered and washed. The residue was repulped to 40% solids and cyanide leached for 24 hours maintaining 0.5 g/L NaCN and pH 10.5 with lime in the presence of 10 g/L activated carbon. The results are summarized in Table 13-11.

Table 13-11 – Effect of Oxidation Temperature

Sample	Conc S^{2-} %	POX Test No.	Pre-acidulation H_2SO_4 Add'n, kg/t		S^{2-} oxid'n in POX %	Hot Cure h	Carbon-in-leach Results					
			fresh	recycled			Test No.	Reag. Cons., kg/t NaCN	Reag. Cons., kg/t CaO	% Extr'n Au	Residue Au. g/t	Head (calc) Au, g/t
A Zone	6.54	POX21	121.2		99.3	0	CIL21	0.32	53.2	98.1	0.3	17.4
		POX27	107.8		99.4	4	CIL27	0.29	2.7	98.3	0.31	16.3
		POX33	58.8		99.4	4	CIL33	0.24	19.5	98.1	0.41	17.9
		POX36		58.8	99.4	4	CIL36	0.2	5.5	98.1	0.38	17.8
South Zone	4.15	POX22	99.1		99.0	0	CIL22	0.22	4.1	98.9	0.12	10.4
		POX28	94.8		99.3	4	CIL28	0.26	4.5	98.5	0.14	9.7
		POX38		47.4	99.3	4	CIL38	0.14	1.5	97.8	0.22	9.4
RW Zone	8.75	POX24	98.1		99.4	0	CIL24	0.46	53	97.9	0.3	15.3
		POX30	96.7		98.9	4	CIL30	0.26	3.6	97.1	0.44	14.3
		POX39		48.4	99.3	4	CIL39	0.14	2	97.8	0.38	14.5
Donchester N	6.38	POX23	167.1		98.8	0	CIL23	0.35	21.9	96.2	0.35	9.1
		POX29	157.4		99.1	4	CIL29	0.19	3.1	97.8	0.22	9.6
		POX37		100	98.8	4	CIL37	0.16	6.7	98.1	0.18	9.2
Donchester S	6.19	POX25	129.7		99.1	0	CIL25	0.29	18.3	97.9	0.16	7.5
		POX31	119.8		98.8	4	CIL31	0.26	4.4	97.5	0.18	6.6
		POX40		59.6	99.1	4	CIL40	0.13	1.7	97.7	0.16	6.5
Duparquet Central	4.8	POX26	38.6		99.0	0	CIL26	0.22	8	94.5	0.43	7.5
		POX32	34.7		97.2	4	CIL32	0.28	3.1	95.5	0.38	8.5
		POX41		17.4	99.1	4	CIL41	0.1	1.4	96.4	0.34	8.8

The results showed that lowering the POX temperature to 210°C did not significantly affect gold recovery. All further pressure oxidation testwork was performed at this reduced temperature.

Subsequent testing was conducted to reduce reagent costs for the POX-CIL flowsheet. The high lime consumption for most of the samples was indicative of the formation of basic iron sulphate (BFS) during the oxidation process. BFS slowly breaks down at the high pH during CIL, generating acid which requires neutralization. In order to overcome this, a hot cure stage was added. In the hot cure, the autoclave discharge was maintained at 95°C to allow the basic iron sulphate to re-dissolve at a much faster rate. After 4 hours, the pulp was filtered and washed and the solids were repulped for CIL as before. A test was conducted on each sample with the addition of the hot cure stage.

Additional tests were performed to reduce the acid requirement in the pre-acidulation stage. In tests POX33 and POX34 conducted on the A Zone concentrate, the fresh acid addition was reduced. Then the effect of recycling acid from the hot cure solution to replace the fresh acid requirement was investigated. Based on the results of POX-CIL33, the addition of acid was reduced by 50% up to a maximum of 60 kg/t and the required acid addition was provided by recycling hot cure solution from a previous test on the same sample. The results are summarized in Table 13-12 and Table 13-13.

The addition of the hot cure stage was effective in reducing the lime requirement in CIL. Lime consumption was decreased by as much as 95% (A Zone) after hot curing. The recovery of gold remained essentially the same and ranged from 95.5% to 98.5% (based on concentrate).

Sulphide oxidation was high in all tests except POX34. Reducing the acid addition by 50% up to 60 kg/t did not significantly affect gold recovery. Using the hot cure solution as the source of acid for pre-acidulation was effective for all samples eliminating the need for fresh acid. Reducing acid addition further to 75% in test POX34 conducted on the A Zone concentrate did not result in adequate destruction of carbonates during pre-acidulation so that oxidation of the sulphides was not achieved under the conditions applied.

Cyanidation of batch pressure oxidation tests (e.g., CN5-6) yielded better gold recovery of ~98% and sulphur oxidation >99% than the pilot plant operation. Visual observations indicated that the pilot plant product contained more basic iron sulphate and jarosite and less hematite than the lab tests. The reason for the slightly lower gold recovery in the tests conducted on the pilot plant product may be due to differences in the precipitate produced in the pilot plant compared to the batch pressure oxidation tests. The acidity of the solution and the concentration of cations such as Na+, and K+ will impact whether hydrolysis reactions favour haematite or jarosite formation.

There is insufficient data to determine the relationship between sulphide oxidation and gold recovery and whether full sulphide oxidation is required to recover the majority of the gold. Earlier batch testwork showed a direct relationship between sulphide oxidation and gold extraction. Hence, a high overall sulphide oxidation target was maintained.

Table 13-12 – Summary of Pressure Oxidation and Hot Cure Results

Sample	Conc S^{2-} %	POX Test No.	Pulp Density % solids	Pre-acidulation			S^{2-} Oxid'n in POX %	POX Solution Assays					POX Residue Assays			Hot Cure h		HC Solution	HC Residue	
				pH	H_2SO_4 Add'n, kg/t			emf mV	Free acid g/L	As mg/L	Fe mg/L	Fe^{2+} mg/L	As %	S^{2-} %	SO_4^{2-} %		Free acid g/L	Fe mg/L	SO_4^{2-} %	Fe %
					fresh	recycled														
A Zone	6.54	POX21	34	1.8	121		99.3	552	48	296	7900	53	0.23	0.05	15.0	0				
		POX27	35	1.8	108		99.4	551	45	443	15000	195	0.22	<0.05	16.2	4	31	29900	14.0	3.2
		POX33	35	3.8	58.8		99.4	526	42	369	13200	268	0.26	<0.05	12.0	0				
		POX34	35	7.2	29.4		35.6	212	0	<1	<1	<1	0.20	4.39	8.0	4	0	<1	7.1	7.0
		POX36	35	3.8		58.8	99.4	539	48	377	15900	176	0.26	0.05	10.0	4	34	30100	9.8	6.0
South	4.15	POX22	41	1.8	99.1		99.0	550	48	517	5330	33	0.23	0.08	8.0	0				
		POX28	42	1.8	94.8		99.3	564	44	439	6730	43	0.25	<0.05	10.2	4	38	12200	10.4	3.2
		POX38	42	2.7		47.4	99.3	530	37	123	5110	119	0.37	<0.05	7.0	4	29	10200	8.7	6.6
RW Zone	8.75	POX 24	30	1.8	98.1		99.4	531	48	246	7790	946	0.29	0.06	16.0	0				
		POX 30	30	1.8	96.7		98.9	542	44	352	11400	82	0.27	0.12	13.0	4	36	21100	11.0	7.3
		POX 39	30	2.7		48.4	99.3	527	45	342	14900	369	0.34	0.08	10.0	4	35	21100	10.0	9.8
Donchester	6.38	POX 23	41	1.8	167		98.8	447	52	285	17000	465	0.09	0.09	17.0	0				
N		POX 29	42	1.8	157		99.1	510	46	244	16100	332	0.11	0.07	15.0	4	34	30100	15.0	3.4
		POX 37	42	3.6		100	98.8	517	43	174	18700	351	0.17	0.09	13.0	4	28	25900	14.0	7.3
Donchester	6.19	POX 25	38	1.8	130		99.1	490	49	681	12600	117	0.45	0.07	11.0	0				
S		POX 31	39	1.8	120		98.8	512	40	1030	17900	388	0.40	0.09	9.9	4	34	24700	9.0	5.6
		POX 40	39	2.6		59.6	99.1	506	38	432	11900	214	0.59	0.07	NA	4	29	18000	3.8	8.6
Duparquet	4.80	POX 26	38	1.8	38.6		99.0	500	42	164	5300	56	0.08	0.06	4.7	0				

Sample	Conc S^{2-} %	POX Test No.	Pulp Density % solids	Pre-acidulation			S^{2-} Oxid'n in POX %	POX Solution Assays					POX Residue Assays			Hot Cure h		HC Solution		HC Residue
				pH	H_2SO_4 Add'n, kg/t			emf mV	Free acid g/L	As mg/L	Fe mg/L	Fe^{2+} mg/L	As %	S^{2-} %	SO_4^{2-} %		Free acid g/L	Fe mg/L	SO_4^{2-} %	Fe %
					fresh	recycled														
		POX 32	40	1.8	34.7		97.2	539	39	283	8470	78	0.08	0.15	4.9	4	36	11300	4.2	4.63
		POX 41	40	2.7		17.4	99.1	527	39	140	6860	49	0.10	0.05	NA	4	38	9020	3.3	5.38

Table 13-13 – Summary of Cyanidation Tests After Pressure Oxidation

Sample	POX Test No.	Pre-acidulation			S^{2-} Oxid'n in POX %	Hot Cure h	Carbon- in-leach Results							
		pH	H_2SO_4 Add'n. kg/t				Test No.	Reagent Add'n., kg/t		Reagent Cons., kg/t		% Extr'n Au	Residue Au. g/t	Head (calc) Au. g/t
			fresh	recycled				NaCN	CaO	NaCN	CaO			
A Zone	POX21	1.8	121		99.3	0	CIL 21	1.21	53.21	0.32	53.2	98.1	0.30	17.4
	POX27	1.8	108		99.4	4	CIL 27	0.98	2.77	0.29	2.7	98.3	0.31	16.3
	POX33	3.8	58.8		994	0	CIL 33	0.99	19.57	0.24	19.5	98.1	0.41	17.9
	POX34	7.2	29.4		35.6	4	CIL 34	0.97	0.88	0.23	0.8	31.0	13.0	18.8
	POX36	3.8		58.8	99.4	4	CIL 36	0.84	5.53	0 .20	5.5	98.1	0.38	17.8
South	POX22	1.8	99.1		99.0	0	CIL 22	0.91	4.10	0.22	4.1	98.9	0.12	10.4
	POX28	1.8	94.8		99.3	4	CIL 28	1.00	4.55	0.26	4.5	98.5	0.14	9.7
	POX38	2.7		47.4	99.3	4	CIL 38	0.87	1.58	0.14	1.5	97.8	0.22	9.4
RW Zone	POX 24	1.8	98.1		99.4	0	CIL 24	1.33	53.05	0.46	53.0	97.9	0.30	15.3
	POX30	1.8	96.7		98.9	4	CIL 30	0.99	3.68	0.26	3.6	97.1	0.44	14.3
	POX 39	2.7		48.4	99.3	4	CIL 39	1.00	2.02	0.14	2.0	97.8	0.38	14 .5
Donchester	POX 23	1.8	167		98.8	0	CIL 23	0.99	21.91	0.35	21.9	96.2	0.35	9.1
N	POX 29	1.8	157		99,1	4	CIL 29	0.96	3.16	0 .19	3.1	97.8	0.22	9.6
	POX37	3.6		100	98.8	4	CIL 37	0.80	6.74	0.16	6.7	98.1	0.18	9.2
Donchester S	POX25 POX 31	1.8 1.8	130 120		99.1 98.8	0 4	CIL 25 CIL 31	1.01 1.00	18.34 4.40	0.29 0.26	18.3 4.4	97.9 97.5	0.16 0.18	7.5 6.6
	POX40	2.6		59.6	99.1	4	CIL 40	0.97	1.76	0.13	1.7	97.7	0.16	6.5
Duparquet	POX26	1.8	38.6		99.0	0	CIL 26	0.98	8.00	0.22	8.0	94.5	0.43	7.5

Sample	POX Test No.	Pre-acidulation			S^{2-} Oxid'n in POX %	Hot Cure h	Carbon- in-leach Results							
		pH	H_2SO_4 Add'n. kg/t				Test No.	Reagent Add'n., kg/t		Reagent Cons., kg/t		% Extr'n Au	Residue Au. g/t	Head (calc) Au. g/t
			fresh	recycled				NaCN	CaO	NaCN	CaO			
	POX 32	1.8	34.7		97.2	4	CIL 32	0.99	3.15	0.28	3.1	95.5	0.38	8.5
	POX 41	2.7		17.4	99.1	4	CIL 41	0.88	1.43	0.10	1.4	96.4	0.34	8.8

Table 13-14 – Summary of Batch Cyanidation Tests on Hot Cure Discharge

Test No	Feed		POX residence time Minutes	Lime boil Min	Pulp density % solids	Reagent Add'n NaCN	CaO kg/t*	Reagent Concs NaCN	CaO kg/t*	Extraction Au	Ag %	Residue assays Au	Ag g/t	S %	Calculated head Au	Ag g/t
CN5	Hot cured autoclave residue	Lab	90	0	36	1.37	4.55	0.24	4.53	98.9	…	0.37	…	0.12	27.1	…
CN6		Lab	90	0	36	1.19	5.44	0.2	5.39	98.4	…	0.5	…	0.08	26.2	…
CN9		Pilot	90	0	35	1.08	7.86	0.17	7.84	96.5	…	0.84	…	0.38	24.3	…
CN10		Pilot	90	0	40	1.01	4.41	0.20	4.34	96.2	…	0.97	…	0.14	25.8	…
CN11		Pilot	90	0	44	0.72	4.08	0.13	4.01	96.0	...	1.01	…	0.43	25.1	…
CN12		Pilot	90	0	39	0.89	3.83	0.10	3.74	96.0	...	1.02	…	0.22	25.8	…
CN13		Pilot	60	0	45	0.96	4.91	0.08	4.83	95.5	...	1.17	…	0.72	26.4	…
CN14		Pilot	60	0	33	1.10	4.57	0.13	4.47	95.5	9.2	1.27	32.8	0.43	28.5	36.3
CN15		Pilot	60	120	25	1.47	119	0.16	119.0	98.1	95.5	0.52	1.6	0.08	31.8	41.2
CN16	Thk u/f shutdown	Pilot	60	0	35	0.85	3.97	0.05	3.92	96.0	…	1.22	…	0.11	30.5	…
CN17	HC4 pump tank out	Pilot	60	0	39	0.84	3.24	0.35	3.19	94.7	…	1.19	…	0.27	22.5	…
CN23	PP HC Thk u/f			0	30	1.24	4.1	0.23	4.0	96.4	9.3	1.06	32.4		28.6	35.4

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Test No	Feed		POX residence time Minutes	Lime boil Min	Pulp density % solids	Reagent Add'n NaCN		Reagent Concs NaCN		Extraction Au		Residue assays			Calculated head Au	
							CaO kg/t*		CaO kg/t*		Ag %	Au	Ag g/t	S %		Ag g/t
CN24	PP HC Thk u/f			120	30	1.41	110.2	0.25	110.1	98.6	95.3	0.36	1.5		28.4	34.6
CN25	PP HC Thk u/f			75	30	1.35	78.6	0.42	78.6	98.3	89.9	0.47	3.4		28.1	35.7

13.1.6.3 Pilot Plant Testwork

SGS Project Reference No. 13054-004– Report 2 [7] summarizes the pressure oxidation pilot test results which consist of feed preparation, pressure oxidation, a hot curing stage and thickening of the final hot cured autoclave residue.

Pressure oxidation feed (flotation concentrate)

The feed for the pressure oxidation testwork was produced from a blended mixture of concentrate produced from an earlier flotation pilot plant [6] presented in above section 13.1.4.3. The composition of the concentrate is listed in Table 13-14.

Table 13-15 – Concentrate Composition

POX Feed Solids					
Au g/t	25.4	Bi g/t	<20	Pb g/t	234
Ag g/t	32	Ca g/t	22,800	Sb g/t	64
S%	16.9	Cd g/t	<10	Se g/t	<30
S2-%	16.8	Co g/t	113	Sn g/t	<20
Fe %	17.1	Cr g/t	162	Sr g/t	323
Cu %	0.041	Cu	330	Ti g/t	4,890
Hg %	39.1	Fe g/t	179,000	Tl g/t	<30
As %	0.69	K g/t	43,900	U g/t	<40
SiO2 %	35.7	Li g/t	<5	V g/t	177
C (t)	1.11	Mg g/t	5,790	Y g/t	38.9
CO3 2- %	4.35	Mn g/t	745	Zn g/t	214
C (g) %	0.02	Mo g/t	207	Cl g/t	100
Al g/t	55,200	Na g/t	5,080	F %	0.069
Ba g/t	1140	Ni g/t	96	Hg g/t	39.1
Be g/t	2.34	P g/t	844	Te g/t	<50

The flotation concentrate was generated from a composite sample produced from drill core from across the deposit. The selected drill core and the average flotation feed mineralized material grade (Au) were selected to be as representative as possible. The details of sample selection are summarized in the SGS report. The head grade of the blended concentrate is 25.4 g/t gold, 32 g/t silver with sulphide grade at 16.8%. The sulphide content is sufficient to operate the pressure oxidation process under autothermal conditions without the requirement for extra heat. The high carbonate content (4.35%) necessitates an acid pre-treatment process to remove carbonate before pressure oxidation to ensure no build-up of carbon dioxide pressure in the autoclave.

No mineralogy of the concentrate feed was reported but based on the chemical composition the feed was calculated to have the mineralogical composition listed in Table 13-16.

Table 13-16 – Calculated Mineral Composition of Concentrates

Mineral	%	Mineral	%
Quartz	11.4	Calcite	2.4
K-feldspar	25.5	Ankerite	2.4
Albite	5.8	Dolomite	2.4
Pyrite	30.8	Kaolinite	3.7
Arsenopyrite	1.5	Fluoroapatite	1.8
Chalcopyrite	0.1	Biotite	9.0
Haematite	1.5	Gypsum	0.8
Rutile	0.8		

Pressure oxidation pilot plant tests

The pressure oxidation tests were carried out using the SGS Lakefield 40L horizontal high pressure titanium vessel. The autoclave was operated and maintained at a temperature and oxygen overpressure of 210°C and 700 kPa respectively. The operating conditions at 60- and 90-minutes residence times were evaluated. The oxygen addition rate equated to 0.45 and 0.48 t of O_2/t_{conc} for 90 and 60 minutes runs respectively (i.e. 2.7 and 2.9 t/t of sulphur in the feed) which is ~40% in excess of the stoichiometric amount.

The feed density was maintained at 18% solids by premixing the concentrate with recycled acid solution. Recycled solution from the hot cure stage at 0.55 L/kg solid, containing 20 g/L Fe^{3+} and 40 g/L free sulphuric acid, was used and found to be sufficient to neutralize the carbonate in the feed concentrate. The carbonate content in the feed concentrates reporting to the autoclave ranged from <0.05 to 0.64%. A summary of the conditions and results are shown in Table 13-17. The increase in concentration from the final autoclave chamber to the autoclave discharge was attributed to concentration in the evaporation/flashing stage and some basic iron salt redissolution.

Table 13-17 – Summary of Pressure Oxidation Conditions and Oxidation

Parameters	Values	
Temperature (°C)	210	
Oxygen overpressure (kPa)	700	
Feed density (%)	18-19	
Retention time (minutes)	60 and 90	
Oxygen addition (O_2 t/t of S)	2.9 and 2.7	
	Sixth Autoclave chamber	**Autoclave discharge values**
Solids		
Fe (%)	14.6	13.9
S (%)	6.8	6.4
% S oxidation	97.9	99.0
Liquor		
Total Fe (mg/L)	5,363	11,490
Ferrous ion (mg/L)	375	475
Arsenic (mg/L)	245	923
Free acid (g/L)	45	46

The reaction profile through the autoclave showed a similar trend for both 90- and 60-minute retention times but with higher sulphide oxidation taking place earlier in the autoclave for the slower throughput (90 minutes). In both, sulphide oxidation was largely complete after the third chamber of the 6-chambered autoclave. The majority of ferrous iron was converted to ferric after the sixth chamber. Overall, a 60-minute retention time appears to be sufficient to achieve a high level of oxidation.

The autoclave discharge liquor had on average 47 g/L free sulphuric acid and contained ~11-12 g/L iron (>95% as Fe^{3+}). Arsenic in solution was on average < 1 g/L. The Fe/As ratio is sufficiently high in the POX liquor for stabilisation of arsenic as a ferric arsenate product following neutralization. The POX liquor also contained on average 1.7 g/L Al, 1.1 g/L Mg and 0.7 g/L Si. There appeared to be no obvious dissolution of chloride from the concentrate to cause any detrimental effect on both pressure oxidation reactions and gold deportment, with levels reported in the liquor of 23 mg/L largely from the Lakefield river water used to pulp the concentrate. Comparisons of the feed and autoclave residue are provided in Table 13-18 and Table 13-19.

X-ray diffraction studies confirmed the beige, yellowish autoclave discharge product was predominately a jarosite product with little haematite formed. The high free acid accompanied by the dissolution of some phyllosilicate and framework silicate minerals promoted jarosite/alunite formation. The autoclave internals were generally clean indicating scaling was not a major issue.

No work was carried out to optimize the pressure oxidation operating conditions apart from retention time. The merits and amount of acid recycled to neutralize the carbonate in the feed to the autoclave needs to be investigated where lowering the free acidity in the autoclave may promote precipitation of hematite and control the amount of jarosite formed. The trade-off of this approach will be that increased carbonate in the feed to the autoclave will yield more carbon dioxide and reduce the oxygen partial pressure in the autoclave and increase the autoclave venting rate. An increased venting rate will lead to lower oxygen efficiency and higher rates of heat (steam) loss and hence are to be avoided if possible.

Table 13-18 – Autoclave Feed and Discharge Compositions for Major Components

	Feed concentrate	Solid residue	Filtrate from residue
SG	2.7	2.93	1.086
Free acid (g/L)	-	-	47
	g/t	g/t	mg/L
Au	25.4	26.3	<0.03
Ag	32	33.7	<0.01
	%	%	mg/L
S	16.9	6.41	-
S^{2-}	16.8	0.13	-
SO_4	-	18.3	-
Fe	17.1	13.83	11873
Fe^{2+}	-	-	489
Cu	0.041	0.036	91
As	0.69	0.4	937
SiO_2	35.7		
Si	-	8.1	713
$C_{(t)}$	1.11	0.13	-
CO_3	4.35	<0.06	-
C_{org}	0.02	0.13	-

Table 13-19 – Autoclave Feed and Discharge Compositions for Other Components

	Feed concentrate	Solid residue	Filtrate from residue
SG	2.7	2.93	1.086
Free acid (g/L)	-	-	47
	g/t	g/t	mg/L
Au	25.4	26.3	<0.03
Ag	32	33.7	<0.01
	%	%	mg/L
S	16.9	6.41	-
S^{2-}	16.8	0.13	-
SO_4	-	18.3	-
Fe	17.1	13.83	11873
Fe^{2+}	-	-	489
Cu	0.041	0.036	91
As	0.69	0.4	937
SiO_2	35.7		
Si	-	8.1	713
$C_{(t)}$	1.11	0.13	-
CO_3	4.35	<0.06	-
C_{org}	0.02	0.13	-

Hot cure

The autoclave discharge slurry was pumped directly to the first of four insulated hot cure tanks. The hot cure was initially operated with four tanks providing a total retention time of 4 hours. The latter half of the pilot operated with five tanks having a total retention time of 5 hours.

The operating conditions, iron species content, as well as free acid assays for hot cure feed (AC Flash) and hot cure discharge (HC#4), are summarised in Table 13-20. The target temperature was maintained at 95 °C with steam heated submerged heating coils within tanks.

Iron in the hot cure discharge was ~17 g/L during the 90-minute autoclave retention time period and then increased to ~20 g/L when the retention time was reduced to 60 minutes. Arsenic content remained consistent throughout at an average of ~1.7 g/L. The total sulphur grade of the hot cure solids was consistently about 6% and close to the autoclave discharge slurry average of 6.4%.

For the basic ferric sulphate dissolution, free sulphuric acid is consumed and soluble ferric iron concentration would be expected to increase. However there appeared to have been little evidence of re-dissolution of basic ferric sulphate, with the majority of conversion having taken place within the autoclave flash let down. Changes in concentrations between autoclave discharge and hot cure discharge appear largely related to the flash letdown redissolution of basic iron salts and some loss of solution through evaporation.

Table 13-20 – Hot Cure Operating Conditions, Average Iron, Arsenic and Free Acid Levels

Parameters	Values	
Temperature (°C)		95
Pulp density (kg/L)		1.20-1.25
Retention time (hrs)		4-5
ORP (mV)		700
	Autoclave discharge	**Final hot cure tank discharge**
$Fe_{(tot)}$ (mg/L)	11,490	19,050
Fe^{2+} (mg/L)	475	663
Fe^{3+} (mg/L) (difference)	11,015	18,387
As (mg/L)	923	1,818
Free Acid (g/L)	46	44

Optimization of the hot cure conditions of temperature and retention time needs to be carried out in conjunction with further pressure oxidation studies.

Hot cure discharge thickening

The hot cure discharge slurry was diluted to ~5-10% solids by recycling thickener overflow and injected in-line with flocculant Magnafloc 455. Delivery into the thickener allowed for a suitable mixing and contact time with the flocculant agent. The hot cure thickener appeared to operate well with the solids settling rate reported to be fast.

Table 13-21 – Thickener Parameters

Parameters	Values
Flocculant	Magnafloc 455
Flocculant dosage (g/t of thickener solids)	95
Settling rate	Reasonably fast (not measured)
Average density of thickener underflow (w/w%)	45
Total suspended solids in overflow (g/L)	2.5
Overflow clarity	Poor

The Flocculant Magnafloc 455 dosage averaged ~95 gram per tonne of thickener feed solids basis. The thickener overflow liquor was not clear and contained total suspended solids of ~2.5 g/L.

Batch cyanidation test for extracting gold and silver

The recovery of gold and silver from the pilot plant hot cure discharge was determined in bottle roll leach tests. A summary of results, including results from two batch pressure oxidation laboratory tests carried out before the pilot plant, are shown in Table 13-14. Pulp samples from the hot cure discharge were taken from the pilot plant every 6 hours. The collected pulp was filtered, washed and then repulped with fresh water for cyanidation. The cyanide concentration was maintained at 0.5 g/L NaCN and the pH maintained at 10.5-11 with lime addition over the 24-hour leach period.

The recovery of gold from the pilot plant hot cure discharge ranged from 94.7% to 96.5% with gold recovery slightly higher for tests conducted after 90-minute POX time (96.2%, Au in residue 0.96 g/t) than those tests conducted after the 60-minute POX time (95%, 1.21 g/t Au in residue). No testwork was conducted to determine the nature of the gold losses in the pilot plant residue samples.

Lime boil was conducted at 95 °C for 4 hours with the addition of 120 kg/t CaO for two tests (CN15, 24) and at 75 kg/t CaO in a third test (CN25) to break down the jarosite and render silver recoverable by cyanidation. The addition of the lime boil resulted in an increase in the silver recovery from 9.2% to 95.5% as well as an increase in gold recovery from 95.5% to 98.1%. Reducing the lime dosage below 75 kg/t resulted in a reduction in gold and silver extraction. Optimum dosage appeared to be 75 kg/t of lime.

Cyanidation of the products of batch pressure oxidation tests (e.g., CN5-6) yielded better gold recovery of ~98% and sulphur oxidation >99% than the pilot plant operation. Visual observations indicated that the pilot plant product contained more basic iron sulphate and jarosite and less hematite than the lab tests. The reason for the slightly lower gold recovery in the tests conducted on the pilot plant product may be due to differences in the precipitate produced in the pilot plant compared to the batch pressure oxidation tests. The acidity of the solution and the concentration of cations such as Na^+, and K^+ will impact whether hydrolysis reactions favour hematite or jarosite formation. The continuous autoclave operation is also “seeded” with iron precipitates whereas the batch test is unseeded.

There is insufficient data from the pilot plant operation to determine the relationship between sulphide oxidation and gold recovery and whether full sulphide oxidation is required to recover the majority of the gold. Earlier batch testwork showed a direct relationship between sulphide oxidation and gold extraction. Hence, a high overall sulphide oxidation target was maintained.

Conclusions

A preliminary pilot test program investigated pressure oxidation and hot curing processing of a Duparquet flotation concentrate to render precious metals extractable by cyanidation.

The recovery of gold from the pilot plant hot cure discharge ranged from 94.7% to 96.5% with gold recovery slightly higher for tests conducted after 90-minute POX time (96.2%, Au in residue 0.96 g/t) than those tests conducted after the 60-minute POX time (95%, 1.21 g/t Au in residue). No testwork was conducted to determine the nature of the gold losses in the pilot plant residue samples.

The pilot plant operated for 48 hours with no significant problems. The 16.8% sulphide sulphur concentrate feed was oxidized efficiently in the autoclave at 210 °C and oxygen overpressure of 700 kPa, with residual sulphide levels averaging 0.13%. A 60-minute retention time appeared sufficient to achieve a high level of oxidation. Recycling a portion of the acidic hot cure thickener overflow allows for neutralization of carbonates present in the feed concentrate. The autoclave discharge was predominately a jarosite product with little hematite formed. The POX liquor contained 47 g/L free acid, 11 g/L Fe (>95% as ferric iron) and ~1 g/L As. No work has been carried out in the pilot plant to optimize the pressure oxidation operating conditions apart from retention time.

The merits and amount of acid recycled to neutralize the carbonate in the feed to the autoclave needs to be investigated. The lowering of the free acidity in the autoclave may promote precipitation of hematite and control the amount of jarosite formed.

The hot cure was operated at 95°C with autoclave discharge product having a total retention time of either 4 or 5 hours to promote the conversion of any basic iron sulphate formed during pressure oxidation. There appeared to be little evidence of re-dissolution of basic ferric sulphate in the hot cure circuit with the majority of conversion having taken place within the autoclave flash letdown. The re-dissolution that did occur happened relatively quickly. Overall iron in solution increased from ~11 g/L in the autoclave discharge to ~17 g/L in the first hot cure stage and up to ~20 g/L through the hot cure circuit. Arsenic content remained consistent throughout at an average of ~1.7 g/L. Optimization of the size of the hot cure circuit in conjunction with further pressure oxidation studies is required. The cyanidation testwork program has demonstrated that high gold and silver recoveries are obtainable for cyanidation of pressure oxidation residues. The recovery of gold from the pilot plant hot cure discharge from batch testwork ranged from 94.7% to 96.5% with average residue assay from 0.96 g/t to 1.21 g/t Au. Lime (CaO) consumptions ranged from 4 to 8 kg/t with NaCN consumptions between 0.05 and 0.24 kg/t.

Lime boil tests on thickened hot cured discharged material, conducted at 95ºC for 4 hours with the addition of 120 kg/t CaO, were effective in breaking down jarosite and increased silver recovery from 9.2% to 95.5%, and gold recovery from 95.5% to 98.1%. Reducing the lime dosage below 75 kg/t resulted in a reduction in gold and silver extraction, with optimum conditions appear to favour 75 kg/t of lime. Promoting conditions favourable for hematite formation over jarosite in the pressure oxidation would undoubtedly reduce lime consumption in the lime boil step.

Precious metal leaching kinetics, carbon adsorption testwork and process modelling were limited to investigating cyanidation and recovery of gold from hot cured discharge samples. No detailed tests were carried out to determine the leaching and carbon adsorption kinetics for both gold and silver following lime boil processing. The leaching and carbon in pulp circuit was remodelled by Tenova-Bateman using the SIMCIL program (developed in the AMIRA P420 project) to take into account and estimate silver loadings to enable designing the whole precious metal leaching, recovery, elution and electrowinning circuits. The cyanidation circuit was configured with two leach tanks followed by six stages of carbon contacting, similar to the SGS model, with carbon transfer rate increased from 1.5 to 2.5 tpd to take into account silver loading. Further work is required to investigate leaching and adsorption kinetics on lime boiled samples to determine the impact of liberated silver on precious metal leaching and adsorption kinetics to establish the final cyanidation extraction and recovery circuit configuration.

Overall the current design for the process plant is limited to one test program on one concentrate sample. The concentrate was obtained from a blend of samples from across the Project. It is recommended that the impact of concentrate variability be investigated with respect to the mine plan and flotation feed variations. Mineralogical characterization and deportment of gold and silver should also be verified. Following further optimization tests on each unit operation, an integrated pilot plant for the main elements of the circuit should be operated for an extended period of time to confirm the results and also test other unit operations within the proposed flowsheet.

13.1.7 Gold Leaching and Carbon Adsorption Testwork

Leach kinetic tests [6] were carried out to determine the rate of gold leaching on a washed hot cure thickener underflow composite. The leach testwork was conducted by bottle roll tests. The leach was conducted at 35% solids density as directed by Clifton Star. The NaCN concentration was maintained at 0.5 g/L and the pH at 10.5 with lime. Each test was carried out for 48 hours with kinetic subsampling at 1, 2, 4, 8, 12 and 24 hours.

To obtain absorption loading kinetics and capacity data the cyanided leach pulps that were generated during the leach kinetic testing were contacted with predetermined amounts of activated carbon in batch tests and bottle rolled for 72 hours. Solution samples were taken for gold analysis at specific time intervals. The data were modelled using an SGS in house program which uses semi-empirical models developed by Mintek in the 1980s to predict operation and plant performance. Equilibrium isotherms for gold cyanide loading on activated carbon were generated and fitted to a Freundlich isotherm non-linear regression.

Leaching kinetics were fast with gold leaching largely complete within the first two hours as illustrated in Figure 13-1.

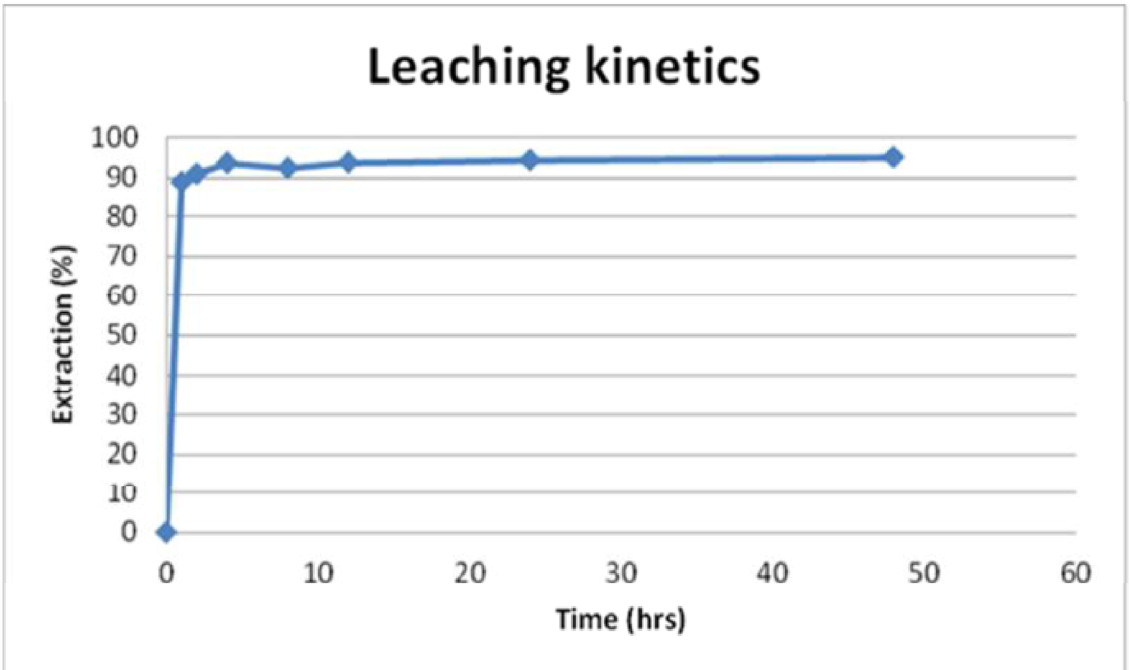


Figure 13-1 – Cyanidation Bottle Roll test showing rapid gold leaching kinetics

The gold adsorption isotherm plot with the predicted curve fit is shown in Figure 13-2. The adsorption equilibrium constant (Freundlich constant 'a') determined for the Duparquet washed solids hot cure thickener underflow was high at 13,513 and is attributed to the high gold grade of the residue and purity of the water.

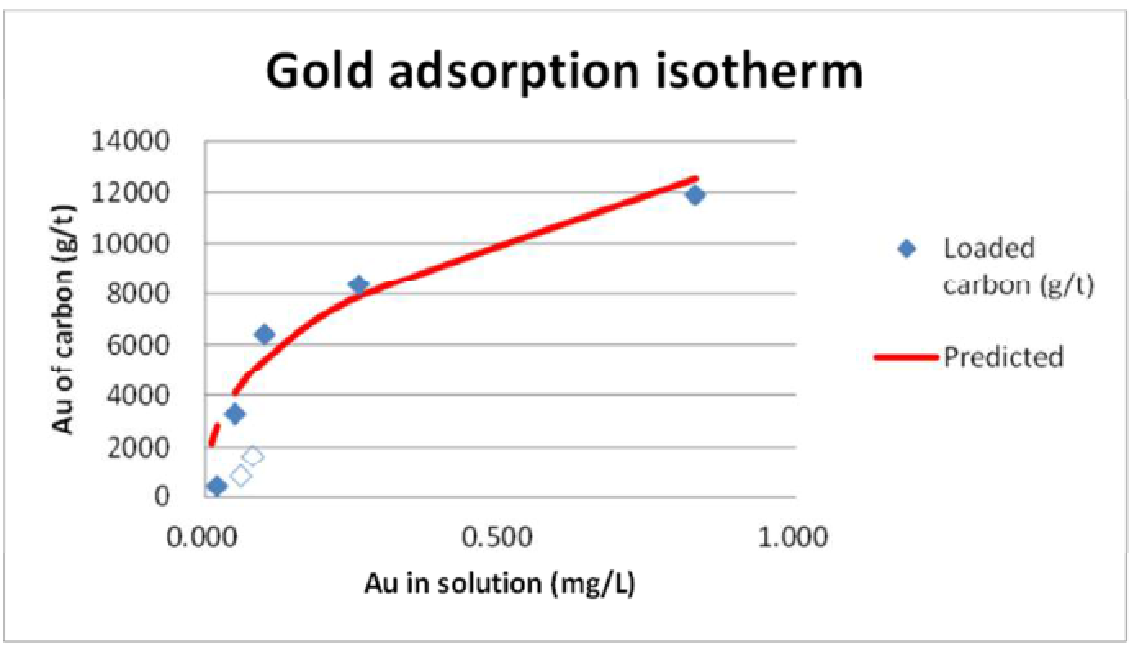


Figure 13-2 – Gold adsorption isotherm

SGS modelled the leaching and adsorption kinetics for gold and proposed a leaching and carbon in pulp configuration for recovering the gold based on the data from the testwork sample, but no data were provided for silver leaching or adsorption kinetics. Based on the test results listed in Table 13-22, extractable gold and silver yields are expected to increase following lime boil from 95% to 98% for gold and from 9% to ≥89% for silver.

The increased extraction of silver will impact the precious metal loading rates and capacity on carbon. Silver loads onto carbon more slowly than gold but elutes and electrowins more rapidly. Hence it will also impact carbon in pulp, elution and electrowinning design.

Table 13-22 – Leach kinetic tests results

Sample	POX Test No.	Pre-acidulation			S^{2-} Oxid'n in POX %	Hot Cure h	Carbon- in-leach Results							
		pH	H_2SO_4 Add'n. kg/t fresh	H_2SO_4 Add'n. kg/t recycled			Test No.	Reagent Add'n., kg/t NaCN	Reagent Add'n., kg/t CaO	Reagent Cons., kg/t NaCN	Reagent Cons., kg/t CaO	% Extr'n Au	Residue Au. g/t	Head (calc) Au. g/t
A Zone	POX21	1.8	121		99.3	0	CIL 21	1.21	53.21	0.32	53.2	98.1	0.30	17.4
	POX27	1.8	108		99.4	4	CIL 27	0.98	2.77	0.29	2.7	98 .3	0.31	16.3
	POX33	3.8	58.8		994	0	CIL 33	0.99	19.57	0.24	19.5	98.1	0.41	17.9
	POX34	7.2	29.4		35.6	4	CIL 34	0.97	0.88	0.23	0.8	31.0	13.0	18.8
	POX36	3.8		58.8	99.4	4	CIL 36	0.84	5.53	0 .20	5.5	98.1	0.38	17.8
South	POX22	1.8	99.1		99.0	0	CIL 22	0.91	4.10	0.22	4.1	98.9	0.12	10.4
	POX28	1.8	94.8		99.3	4	CIL 28	1.00	4.55	0.26	4.5	98.5	0.14	9.7
	POX38	2.7		47.4	99.3	4	CIL 38	0.87	1.58	0.14	1.5	97.8	0.22	9.4
RW Zone	POX 24	1.8	98.1		99.4	0	CIL 24	1.33	53.05	0.46	53.0	97.9	0.30	15.3
	POX30	1.8	96.7		98.9	4	CIL 30	0.99	3.68	0.26	3.6	97.1	0.44	14.3
	POX 39	2.7		48.4	99.3	4	CIL 39	1.00	2.02	0.14	2.0	97.8	0.38	14.5
Donchester	POX 23	1.8	167		98.8	0	CIL 23	0.99	21.91	0.35	21.9	96.2	0.35	9.1
N	POX 29	1.8	157		99.1	4	CIL 29	0.96	3.16	0.19	3.1	97.8	0.22	9.6
	POX37	3.6		100	98.8	4	CIL 37	0.80	6.74	0.16	6.7	98.1	0.18	9.2
Donchester S	POX25 POX 31	1.8 1.8	130 120		99.1 98.8	0 4	CIL 25 CIL 31	1.01 1.00	18.34 4.40	0.29 0.26	18.3 4.4	97.9 97.5	0.16 0.18	7.5 6.6
	POX40	2.6		59.6	99.1	4	CIL 40	0.97	1.76	0.13	1.7	97.7	0.16	6.5
Duparquet	POX26	1.8	38.6		99.0	0	CIL 26	0.98	8.00	0.22	8 .0	94.5	0.43	7.5

Sample	POX Test No.	Pre-acidulation			S^{2-} Oxid'n in POX %	Hot Cure h	Carbon- in-leach Results								
		pH	H_2SO_4 Add'n. kg/t				Test No.	Reagent Add'n., kg/t		Reagent Cons., kg/t		% Extr'n Au	Residue Au. g/t	Head (calc) Au. g/t
			fresh	recycled				NaCN	CaO	NaCN	CaO			
	POX 32	1.8	34.7		97.2	4	CIL 32	0.99	3.15	0.28	3.1	95.5	0.38	8.5
	POX 41	2.7		17.4	99.1	4	CIL 41	0.88	1.43	0.10	1.4	96.4	0.34	8.8

The leaching and carbon in pulp circuit were remodelled by Tenova Mining & Minerals–Bateman Engineering Pty Ltd ("Tenova-Bateman") using the SIMCIL program developed in the AMIRA P420 project to take into account silver. The throughput and pulp density have also been adjusted in the SIMCIL model to reflect expected conditions in the commercial plant. Both SGS and SIMCIL model conditions are presented in Table 13-22.

Gold and silver leach rates were assumed to be similar at 0.8 t/h.g, and an adsorption rate of 0.010/h. Silver leaching rates are normally slower than gold but as the silver is liberated from jarosite following lime boil it may be assumed that leaching will take place at a similar rate to gold. The Freundlich exponent constant 'a' of 13,513 determined by SGS was used and the 'b' value set at 1.0 for silver modelling. The SIMCIL modelled output is shown in Table 13-23. The configuration of two leach tanks followed by six stages of carbon contacting was used in the SIMCIL model, similar to that utilised in the SGS model. The increased precious metal loadings on carbon necessitated almost doubling the carbon transfer rate from 1.5 to 2.5 tpd.

Further work is required to investigate leaching and adsorption kinetics on lime boiled samples to determine the impact of liberated silver on precious metal leaching and adsorption kinetics to ensure cyanidation extraction and recovery circuit configurations are correctly designed.

Table 13-23 – Parameters modelling cyanide leach and adsorption circuit

Inputs	SGS optimized regime	Modified by Tenova-Bateman for throughput and silver extraction
Slurry feed rate (m^3/h)	45.9	50.5
Solids (t/h)	21.5	32
Solution (m^3/h)	39.9	38.7
Consider leach after carbon addition	N	N
Ore head grade Au (g/t)	26.5	17.2
Ore head grade Ag (g/t)		28.8
Gold on stripped carbon (g/t)	50	50
Silver on stripped carbon (g/t)		250
Adsorption tank(s) size	120	154
Carbon frequency advance (% in 24 hours)	50%	73%
Leaching		
Au leached	94.6%	98.0%
Ag leached		89.0%
Leach time before carbon addition (h)	24	24
Leach only total tankage (m^3)	1,102	1,290
Number of leaching tanks	2	2
Volume of leaching tanks (m^3)	551	645
CIP		
Number of stages	6	6
Total CIP/CIL volume (m^3)	720	924
Slurry residence time in each adsorption tank (h)	2.6	3.1
Gold grade in residue (g/t)	1.45	0.29
Silver grade in residue (g/t)		2.91
Gold in final barren solution (mg/L)	0.004	0.005
Silver in final barren solution (mg/L)		0.064
Gold in loaded carbon (g/t)	8,838	4,630
Silver in loaded carbon (g/t)		7,273

Inputs	SGS optimized regime	Modified by Tenova-Bateman for throughput and silver extraction
Carbon residence time/stage (h)	48	33
Carbon Concentration (g/L Solution)	25	25
Equivalent transferred carbon unit flowrate (kg/h)	63	117
Daily carbon transfer / batch elution capacity (kg/d)	1,500	2,800
Carbon Inventory per stage (kg)	3,000	3,850
Carbon inventory all stages (tons)	18	23.1
Gold on barren carbon (g/t)	34	50
Silver on barren carbon (g/t)		260
CIP gold recovery per day (g/d)	13,183	12,964
CIP silver recovery per day (g/d)		20,364
Au&Ag in loaded carbon / Au&Ag in feed	327	259

Table 13-24 – Output of modelling leaching and adsorption circuit with SIMCIL program

Stage	Gold				Silver			
	Solution Au (g/t)	Carbon Au (g/t)	Ore Au (g/t)	Gold ext'n (%)	Solution Ag (g/t)	Carbon Ag (g/t)	Ore Ag (g/t)	Silver ext'n (%)
Feed	0	-	17.15	-	0	-	28.8	-
1	13.112	-	1.52	-	20.431	-	4.45	-
2	13.892	-	0.59	-	21.434	-	3.25	-
3	1.653	4,630	0.49	86	7.038	7,273	3.13	60
4	0.212	665	0.44	72	2.308	2,698	3.07	49
5	0.033	144	0.41	36	0.736	1,049	3.03	33
6	0.01	72	0.38	11	0.250	495	3.01	15
7	0.006	59	0.37	5	0.107	320	2.99	6
8	0.005	53	0.36	3	0.064	266	2.98	2

13.1.8 Cyanide Destruction

13.1.8.1 Bench Scale – September 2012

The SO_2/air method was used to destroy the cyanide in the CIL tailings [2]. Batch tests were conducted at pH 8.5 to lower the CNWAD level in the pulp to approximately 1 mg/L. The required amount of copper was added to catalyse the cyanide oxidation reaction at the start of the test. SO_2 was added continuously as a sodium metabisulphite solution. After 1 hour, the barren solution was sampled to determine the CNWAD level by picric acid. If the analysis was >1 mg/L, the test was continued. The results are presented in Table 13-24.

Table 13-25 - Summary of cyanide destruction tests

Sample	Test No.	Retention Time h	Reagent Addition						CND Barren Solution			
			g/g CNWAD			g/L Feed Pulp			CNT mg/L	CNWAD mg/L	Cu mg/L	Fe mg/L
			SO_2	CaO	Cu	SO_2	CaO	Cu				
A Zone	CND36	60	4.75	0	0.04	0.50	0	0.004	6.45	1.09	0.17	2.08
South RW	CND38	180	18.58	2.2	0	2.07	0.25	0	1.70	0.13	<0.05	0.69
Zone	CND39	60	6.33	0.8	0.04	0.65	0.08	0.004	0.77	0.77	0.08	0.34
Donchester N	CND37	90	11.88	1.3	0.04	1.04	0.11	0.004	8.25	1.32	0.05	2.70
Donchester S	CND40	120	12.67	1.5	0.03	1.49	0.18	0.004	0.79	0.62	0.26	0.28
Duparquet	CND41	120	12.67	0.3	0.03	1.58	0.04	0.004	0.29	0.22	3.57	0.12

Although the CNWAD level was reduced to <1 mg/L, the total cyanide (CNT) level was significantly higher for the A Zone and Donchester N samples due to the presence of ferrocyanide. It should be noted that batch tests generally require higher reagent additions than required in continuous operation. Continuous testing is required to optimize the cyanide destruction conditions.

13.1.8.2 Pilot Plant

SGS Project Reference No. 13054-004 – Report 4 describes the bench-scale testwork program that was conducted to examine various process options relating to the recovery of gold by carbon adsorption from washed hot cure thickener underflow product. The program also investigated neutralization of hot cure thickener overflow solution and cyanide destruction of the cyanide leach product of the pilot plant flotation tailing and hot cure thickener underflow.

The recovery of gold from washed hot cure thickener underflow product by carbon adsorption and neutralization of the hot cure thickener overflow solution were discussed in 13054-002 report 1. The cyanide destruction testwork and its results are described below.

Cyanide destruction using the SO_2/air process was examined briefly for the cyanidation product of PP-05 and PP-09 flotation tailings. All the samples responded well to the SO_2/air method of cyanide destruction with moderate reagent consumptions.

The results of the tests conducted on the PP-05 rougher tailings showed that the cyanide was effectively destroyed with an SO_2 addition of 5.7 g/g CNWAD (Weak Acid Dissociable Cyanide) with a copper addition as copper sulphate of 0.1 g Cu/g CNWAD. Reducing the copper addition by half resulted in an increase in the CNT (Total Cyanide) although the CNWAD remained similar. The SO_2 addition in the bulk test was slightly higher than targeted because the feed rate was slightly lower than the target. A polishing stage is required to achieve a CNT analysis of less than 1 mg/L bringing the total copper addition back to 0.1 g Cu/g CNWAD.

Similar conditions were applied to the PP-09 rougher tailing. The testwork indicated that this tailing needs a higher copper addition of 0.2 g Cu/g CNWAD to achieve the target CNT analysis. The SO_2 requirement to achieve the target was 5.2 g/g CNWAD. The results of the cyanide destruction test are summarized in the Table 13-25.

Table 13-26 - Results of continuous SO_2/air cyanide destruction tests on CIP barren pulp

Feed/ Test	Test	Pulp Density % solids	Reactor Vol. L	Test Time min	Reten. Time min	Solution Analysis					Reagent Addition								
											g/g CN_{WAD}			g/L Feed Pulp			kg/t Solids		
						pH	CNT mg/L	CNWAD mg/L	Cu mg/L	Fe mg/L	SO_2 Equiv.	CaO	Cu	SO_2 Equiv.	CaO	Cu	SO_2 Equiv.	CaO	Cu
Feed - CN7 Pulp (PP-05 Plot tail)						*10.1*	*266*	*236*	*3.25*	*5.41*									
CND 1-1		50	1	210	64	8.5	0.4	<0.1	<0.05	0.12	5.66	2.12	0.10	0.98	0.37	0.017	1.34	0.50	0.024
CND 1-2	SO_2/air	50	1	180	65	8.5	1.3	<0.1	0.12	0.46	4.91	2.78	0.09	0.85	0.49	0.016	1.16	0.66	0.022
CND 1-3		50	1	180	59	8.5	15.3	0.5	<0.05	3.43	4.42	2.27	0.04	0.76	0.40	0.007	1.04	0.54	0.010
CND 1-4		50	1	180	57	8.5	9.9	0.6	0.10	3.43	3.86	3.22	0.04	0.66	0.57	0.007	0.91	0.76	0.010
CND 1-5	SO_2/air	50	2	390	65	8.5	7.1	<0.1	0.15	1.53	6.86	4.19	0.05	1.18	0.74	0.009	1.62	0.99	0.012
	Polishing	50	20	30	30	8.3	<0.1	<0.1	1.60	0.10			0.04			0.007			0.010
Feed - CN8 Pulp (PP-09 Flot Tail)						*10.0*	*241*	*232*	*4.65*	*6.91*									
CND 2-1		50	1	180	56	8.5	9.1	0.2	0.11	3.14	4.84	3.98	0.09	0.82	0.69	0.015	1.12	0.92	0.020
CND 2-2	SO_2/air	50	1	165	59	8.5	10.0	0.3	<0.05	3.04	3.76	1.74	0.09	0.64	0.30	0.015	0.87	0.40	0.020
CND 2-3		50	1	180	62	8.5	13.4	0.2	0.15	3.25	6.88	3.66	0.10	1.17	0.64	0.017	1.60	0.85	0.023
CND 2-4		50	1	180	59	8.5	4.99	0.3	<0.05	1.65	5.19	3.91	0.14	0.88	0.68	0.023	1.21	0.91	0.032
CND 2-5	SO_2/air	50	1	180	59	8.5	1.65	<0.1	0.08	0.80	5.10	3.50	0.18	0.87	0.61	0.031	1.19	0.81	0.042
	Polishing	50	5	30	30	8.3	0.12	<0.1	0.14	0.06			0.04			0.007			0.010
CND 2-6	SO_2/air	50	1	180	59	8.5	<0.1	<0.1	0.25	0.09	5.18	3.59	0.23	0.88	0.63	0.039	1.20	0.83	0.053
CND 2-7	SO_2/air	50	2	317	61	8.5	0.16	<0.1	<0.05	<0.05	5.24	2.81	0.23	0.89	0.49	0.039	1.22	0.65	0.054
Feed - CN21 Pulp (HC Thk u/f)						*10.1*	*283*	*224*	*2.04*	*0.07*									
CND 3-1		33	1	180	60	8.6	0.37	<0.1	<0.05	0.06	4.48	0.09	0.08	0.85	0.02	0.015	2.04	0.04	0.037
CND 3-2	SO_2/air	33	2	394	62	9.4	0.31	<0.1	<0.05	0.08	4.62	0.00	0.09	0.88	0	0.018	2.10	0	0.043
CND 3-3		33	2	630	61	9.3	0.36	0.1	<0.05	0.06	4.45	0.00	0.09	0.84	0	0.017	2.03	0	0.041

13.1.9 Sedimentation Testwork

SGS Project Reference No. 13054-004 – DRAFT – Report 3 [8] summarizes the rheometallurgical responses (i.e., solid-liquid separation and rheology) of the flotation tailings (PP-05 Ro Tail), flotation concentrate (PP Cl Conc (POX Feed)), hot cure discharge (HC-4 Discharge), and combined leached tailings (Comb CND + Neut Solids) that were produced as part of the pilot plant test program.

Preliminary static settling test results for flotation tailings, identified as "PP-05 Ro Tail", indicated that the sample settled well in the presence of 20 g/t of BASF Magnafloc 333 flocculant, producing a 66% w/w solids underflow from a 10% w/w solids thickener feed. The resulting supernatant was slightly cloudy after 60 minutes of elapsed settling time. Relevant thickener data included: 0.045 m^2/tpd thickener underflow unit area (TUFUA), 0.016 m^2/tpd thickener hydraulic unit area (THUA), and 883 m^3/m^2/d initial settling rate (ISR). The supernatant total suspended solids (TSS) after 1 hour of elapsed settling was 22 mg/L.

Preliminary static settling test results for the flotation concentrate, identified as "PP Cl Conc. (POX Feed)", indicated that the sample settled well in the presence of 33 g/t of BASF Magnafloc 333 flocculant, producing a 65% w/w solids underflow from a 20% w/w solids thickener feed. The resulting supernatant was slightly cloudy after 60 minutes of elapsed settling time. Relevant thickener data included: 0.059 m2/tpd TUFUA, 0.006 m^2/tpd THUA, and 536 m^3/m^2/d ISR. The TSS after 1 hour of elapsed settling was 16 mg/L. Preliminary static settling test results for the hot cure discharge, identified as "HC-4 Discharge", indicated that the sample settled in the presence of 92 g/t of BASF Magnafloc 333 flocculant, producing a 30% w/w solids underflow from a 3% w/w solids thickener feed. The resulting supernatant was still cloudy after 60 minutes of elapsed settling time. Relevant thickener data included: 0.374 m2/tpd TUFUA, 0.075 m^2/tpd THUA, and 562 m^3/m^2/d ISR. The TSS after 1 hour of elapsed settling was 625 mg/L.

Preliminary static settling test results for the combined leached tailing, identified as "Comb CND+Neut Solids", indicated that the sample settled well in the presence of 36 g/t of BASF Magnafloc 333 flocculant, producing a 36% w/w solids underflow from a 2.5% w/w solids thickener feed. The overflow was slightly cloudy after 60 minutes of elapsed settling time. Relevant thickener data included: 0.26 m^2/tpd TUFUA, 0.07 m^2/tpd THUA, and 647 m^3/m^2/d ISR. The TSS after 1 hour of elapsed settling was 16 mg/L. The preliminary static settling test results are summarized in Table 13-27.

Table 13-27 - Preliminary static settling test results

Sample	Pulp pH	Flocculant BASF	Dosage g/t	Feed[1] % wt	U/F[2] % wt	CSD U/F[3] % wt	TUFUA[4] m²/tpd	THUA[5] m²/tpd	ISR[6] m³/m²/d	O/F[7]	TSS mg/L
PP-05 Ro Tail	7.3	Magna-floc 333	20	10	66	70	0.045	0,016	883	S.C	22
PP Cl Conc (POX Feed)	7.9		33	20	65	65	0.06	0.01	536	S.C	16
HC-4 Discharge	0.6		92	3	30	42	0.37	0.08	562	Cloudy	625
Comb CND+Neut Solids	8.4		36	3	36	44	0.26	0.07	647	S.C	16

All values were calculated without safety factor.

1. Autodiluted thickener feed
2 . Ultimate Underflow Density
3. Maximum Underflow Density predicted by Critical Solid Density (CSD)
4. Thickener Underflow Unit Area corrected with actual CSD   All tests are raked at 1.0 r.pm.
5. Thickener Hydraulic Unit Area corrected with actual CSD
6. Initial Settling Rate
7. Clarity 60 minutes into the test

The results of the preliminary static settling-thickening were used for reagent selection, optimization of thickener feed solid density and flocculant dosage. The flocculant screening and selection tests were conducted at 5% w/w solids. Five (5) different BASF Magnafloc products were tested with the best response from the non-ionic flocculant Magnafloc 333. The settling rate was fast but produced a cloudy supernatant. Using a combination of coagulant BASF Magnafloc 1687 with Magnafloc 333 did not improve liquor clarity.

The optimized dynamic settling conditions for PP-05 Ro Tail predicted 0.100 and 0.011 m^2/tpd for thickener underflow (TUFUA) and hydraulic (THUA) unit areas, respectively. This corresponded to 86.5 m^3/m^2/day net rise rate, 0.415 Um% net solids loading, and 3.60 m^3/m^2/h net hydraulic loading. The overflow TSS was 43 mg/L and the underflow solids density at 65.2% w/w solids under these conditions. A 30 mn extended thickening increased the underflow density to 69.3% w/w solids yielding about 23 Pa un-sheared vane yield stress. These results were produced at 10% w/w solid density at a dosage of 25 g/t BASF Magnafloc 333 flocculant and 1.11 hours residence time.

The optimized dynamic settling conditions for PP Cl Conc. (POX Feed) predicted 0.100 and 0.025 m^2/tpd for thickener underflow (TUFUA) and hydraulic (THUA) unit areas, respectively. This corresponded to 35.9 m^3/m^2/d net rise rate, 0.415 t/m^2/h net solids loading, and 1.496 m^3/m^2/h net hydraulic loading. The overflow TSS was 53 mg/L and the underflow solids density at 61.8% w/w solids under these conditions. A 30 mn extended thickening increased the underflow density to 65.5% w/w solids yielding about 36 Pa un-sheared vane yield stress. These results were produced at 20% w/w solid density at a dosage of ~40 g/t BASF Magnafloc 333 flocculant and 1.28 hours residence time.

The overflow stayed cloudy for the entire duration of the test run with TSS consistently exceeding ~1500 mg/L. The TUFUA started at 0.401 and decreased to 0.331 m^2/tpd; THUA started at 0.039 and decreased to 0.037 m^2/tpd. The net rise rate increased from 72.0 to 87.2 m^3/m^2/d. The net solids loading increased from 0.104 to 0.126 t/m^2/h. The net hydraulic loading increased from 3.00 to 3.64 m^3/m^2/h. The underflow solids density reached 41.2% w/w solids at the end of run 1 and 30 mn extended thickening increased the underflow density to 42.6% w/w solids yielding about 25 Pa un-sheared vane yield stress. These results were produced at 3% w/w solid density at a dosage of -100 g/t BASF Magnafloc 333 flocculant.

The optimized dynamic settling conditions for Comb CND+Neut predicted 0.22 and 0.011 m^2/tpd for thickener underflow (TUFUA) and hydraulic (THUA) unit areas, respectively. This corresponded to 145.2 m^3/m^2/d net rise rate, 0.190 t/m^2/h net solids loading and 1.496 m^3/m^2/h net hydraulic loading. The overflow TSS was 73 mg/L and the underflow solids density at 42.3% w/w solids under these conditions. 30 mn extended thickening increased the Run 1 underflow density to 43.2% w/w solids yielding about 29 Pa un-sheared vane yield stress. These results were produced at 3.0% w/w solid density at a dosage of ~40 g/t BASF Magnafloc 333 flocculant and 2.32 hours residence time.

The dynamic settling test results under optimum conditions are summarized in Table 13-28.

Table 13-28 - Dynamic settling test results

Sample	d80, µm	Flocculant BASF	Dosage, g/t	Feed[1], % wt	U/F[2], % wt	U/F Extended, % wt	TUFUA[3], m^2/tpd	TUHUA[4], m^2/tpd	Net Rise Rate, $m^3/m^2/d$	Net Solids Loading, $t/m^2/day$	Net Hydraulic Loading, $m^3/m^2/d$	Res. Time, h Solids vs. UF	Overflow Visual	TSS, mg/L
PP-05 Ro Tail	89	Magna-floc 333	25	10	66.2	69.8	0.091	0.01	96.2	0.5	4.01	1.24	Clear	51
PP Cl Conc (POX Feed)	33		40	20	61.8	65.5	0.1	0.025	35.9	0.4	1.496	1.28	S.C	53
HC-4 Discharge	15		100	3										
Comb CND+Neut Solids	12		40	3	42.3	43.2	0.220	0.011	145.2	0.2	1.50	2.32	S.C	73

All values were calculated without a safety factor. S.C. indicates slightly cloudy.

1. Autodiluted Thickener Feed
2. Ultimate Underflow (UF) Density
3. Thickener Underflow Unit Area
4. Thickener Hydraulic Unit Area

The Critical Solids Density (CSD1) of the PP-05 Ro Tail sample was ~70% w/w solids, which corresponded to a yield stress of 52 Pa under un-sheared flow conditions and 22 Pa under sheared conditions. The CSD of the PP Cl Conc. (POX Feed) thickened underflow sample was -65% w/w, which corresponded to a yield stress of 38 Pa under un-sheared flow conditions and 29 Pa under sheared conditions. The CSD of the HC-4 Discharge thickened underflow sample was ~41.5% w/w, which corresponded to a yield stress of 24 Pa under un-sheared flow conditions, and 8 Pa under sheared conditions. The CSD of the HC-4 Discharge-Washed Solids thickened underflow sample was ~36% w/w., which corresponded to a yield stress of 29 Pa under un-sheared flow conditions and 12 Pa under sheared conditions. The CSO of the Combined CNO+Neut. solids thickened underflow sample was ~43.8% w/w, which corresponded to a yield stress of 30 Pa under un-sheared flow conditions and 17 Pa under sheared conditions.

Conclusions

Overall, the aforesaid rheometallurgical test data materially reflect the liquid-solid separation and flow behaviours of the process samples tested, rendering them suitable to be used as design criteria. A possible exception to this assessment involves the hot cure stream which displayed a relatively complex rheometallurgical response. To overcome the high overflow total suspended solids content a further clarification step to remove fine solids from the supernatant may be required.

Further work is still required to carry out specific CCD washing tests and model the process. The wash ratio and washing efficiency through testwork and modelling needs to be evaluated to accurately design the counter current decantation circuit.

Note: The Critical Solids Density (CSD) value is predictive of the maximum underflow solids density achievable in a commercial thickener and of the underflow solids density and pump-ability ranges achievable in practice and with reasonable friction pressure losses for an economically feasible operation.

13.1.10 Filtration Testwork

The Outotec Filtration Test Report [11] includes the results of the filtration test on the flotation cleaner concentrate using a Larox Pressure Filter to achieve filter cake with moisture content of less than 8%. The testwork evaluated filter cloth selection, filter cake thickness, filtration rate, moisture content of the cake, and cake handling characteristics. The results are indicated in Table 13-29.

Table 13-29 - Flotation concentrate pressure filtration test results

Sample - Unit	pH	Air Drying Time (min)	Filtration Rate (kg/m^2/h)	Filter Cake Moisture (% W/W Water)	Filter Cake Thickness (mm)	Pumping Pressure (Bar)	Pressing Pressure (Bar)	Air Pressure (Bar)
Conc. – Larox 100	8	1-4	699-1097	5-8	46-54	6	12	7-10

13.1.11 Neutralization Testwork

Neutralization tests were performed on the pressure oxidation liquor to remove arsenic [1]. The neutralization solution (pH 8) analysed <0.05 mg/L As. A Toxicity Characteristic Leachate Procedure (TCLP) on the neutralization solids confirmed that the arsenic was successfully removed in a stable form with leachate analyses below 0.03 mg/L As.

Seven neutralization tests [2] were performed to determine the quantity of flotation tailings, limestone ($CaCO_3$) and lime (CaO) required to neutralize the POX & hot cure solution. The results of testwork investigating limestone/ lime and flotation tails/lime combinations are shown in Table 13-30.

Table 13-30 - Results of neutralization testwork on CCD overflow liquors

Test No	Reagent addition								Solution analyses mg/L		% solids w/w
	Limestone			Rougher tails to pH 4			CaO to pH				
	pH	g/L	kg/t	pH	g/L	kg/t	g/L	kg/t	As	Fe	
NT-1	4.9	137	579	-	-	-	7.8	33	0.1	0.07	18.7
NT-2	-	-		4.1	727	3,072	37.4	158	0.11	<0.05	43.2
NT-3	5.1	133	561	-	-	-	8.13	34.3	0.1	0.4	33
NT-5	-	-	-	4.4	832	3,516	27	114	0.18	<0.2	44.1
NT-6	5.6	110	465	-	-	-	8.17	34.5	0.08	0.08	15
NT-7	5.1	106	446**	-	-	-	7.4	31.3	0.08	0.08	19

* kg per tonne of hot cure discharge solids (4.225 L of HC solution per kg of HCD solids)
** 2/3 mine site limestone (38% CO_3) + 1/3 reagent grade limestone (97.6% $CaCO_3$)

A combination of limestone and lime successfully reduced the arsenic and iron level in solution to below 0.1 mg/L. The limestone used in this study was classified as a mine site limestone with a grade of 38% carbonate content. Assuming all the carbonate is available for dissolution this is equivalent to between 176.7 and 220 kg of pure calcium carbonate per t. of hot cure discharge solids. For the final test (NT-7) where a combination of mine site limestone and reagent grade limestone was used, a higher consumption was observed and was equivalent to 258 kg of pure calcium carbonate per t. of hot cure discharge solids.

A combination of flotation tailing and lime additions (NT2 and NT5) was also investigated but it was found to be an ineffective method for neutralizing the CCD (Counter Current Decantation) overflow liquor due to the large quantity of flotation tailing required to reduce the pH to 4.0. In addition, slightly elevated concentrations of As, Mn and Sr were observed in the final effluent neutralized with flotation tailings and lime.

More testwork is required and should be carried out on limestone sourced for the Project. In particular, tests should be performed to determine optimum particle size of the limestone, reaction kinetics and required dosage rates.

After filtering the hot cure product, the solutions for each sample were neutralized using limestone and lime. The purpose was to prepare the neutralized pulp for subsequent environmental studies. $CaCO_3$ was initially added to increase the pH to 4.5. The slurry was conditioned for 60 minutes. Hydrated lime was then added to increase the pH to 8. The test data are shown in Table 13-31.

Table 13-31 – Neutralization test results

Sample	Test No.	Reagent Addition				Solution Analysis*mg/L		Product	
		$CaC0_3$ to pH 4.6		CaO to pH8				Precipitate kg/t	Density % solids
		g/L	kg/t	g/L	kg/t	As	Fe		
A Zone	Neut 36	135	241	12	21	0.02	<0.03	591	25
South RW	Neut 38	65	85	10	13	0.02	<0.03	241	15
Zone	Neut 39	89	169	8	15	0.04	<0.03	476	17
Donchester	Neut 37	106	161	14	22	0.02	<0.03	391	24
Donchester	Neut 40	81	127	11	16	0.05	<0.03	327	17
Duparquet	Neut 41	84	126	7	11	0.03	<0.03	275	16

*solution filtered on 0.45 µm filter paper

The sequential neutralization testwork with limestone and lime successfully demonstrated that the arsenic and iron levels in thickener overflow solutions can be reduced to below 0.1 mg/L. The Fe^{3+}/As ratio in the hot cure discharge liquor is suitably high to favour the effective stabilisation of arsenic as an arsenate upon neutralization.

However, the dosage and source of limestone need to be re-evaluated. The neutralization capacity of flotation tailings is insufficient to be used as a first step neutralization agent and hence was not included as a neutralizing agent in the process design.

13.1.12 Environmental Testwork

13.1.12.1 Preparation of samples for environmental characterization – September 2012

Four samples were prepared for environmental characterization [2]. Three of the samples represented the individual tailing streams namely the flotation tailing, the cyanide destruction barren pulp and the POX neutralized liquor and sludge. The fourth sample represented the combined tailing. It was prepared by combining the individual streams in the appropriate relative amounts based on the metallurgical test results. The details of sample preparation and the results of the environmental tests will be presented and discussed in a separate report and are presented below.

13.1.12.2 Environmental Testwork – November 2012

SGS Project Reference No. 13054-002 – Report 2 includes the environmental characterization of the flotation rougher tailings, cyanide destruction barren pulp, neutralization sludge, and the combined tailings products from the six mineralized material samples. Environmental tests were conducted on the twenty-four pulp samples from metallurgical testwork programs [3].

Test Methods

The following sections provide a brief overview of the test methods included in the environmental characterization program.

X-ray Fluorescence (XRF) Whole Rock Analyses

Whole rock analyses were completed on the samples using XRF in order to determine the elemental concentrations of the major rock forming constituents. This method quantifies major elements present and reports them as oxides to permit a mass balance assessment against the component of a sample that is amenable to oxidization (loss on ignition).

ICP-OES/MS Strong Acid Digest Elemental Analyses

The samples were digested using an acid mixture of HNO3, HF, HCIO4, and HCI to obtain a near total dissolution of the elements being analyzed. ICP-OES/MS trace metal scans were performed to provide quantitative analyses of the elemental components of the sample material. Analyses requested included: Ag, Al, As, B, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Hg, K, Li, Mg, Mn, Mo, Na, Ni, P, Pb, Sb, Se, Si, Sn, Sr, Th, Ti, Tl, U, V, W, Y, and Zn. Mercury analyses were completed by cold vapour atomic absorption spectroscopy (CVAAS).

Toxicity Characteristic Leaching Procedure (TCLP- US EPA Method 1311)

The TCLP was used to determine the mobility of inorganic contaminants present in the waste materials which would be chemically stable and insoluble under the aggressive pH environment imposed by the method.

A brief assessment of the sample's neutralization capacity determines the leachant used in the test. In cases where the sample has limited acid neutralization capacity, TCLP extraction fluid #1 is used. Extraction fluid #1 is a combination of glacial acetic acid and deionised water buffered by sodium hydroxide (resulting pH 4.93 ± 0.05). In cases where the residues have sufficient acid neutralization capacity, TCLP extraction fluid #2 is used. This extraction fluid provides 2 eq/kg of acid through addition of glacial acetic acid to deionised water (resulting pH 2.88 ± 0.05). The leachant is added to the sample at a 20:1 liquid-to-solids ratio and the sample container is rotated end over end at 29 ± 2 rpm for 18 hours. The resultant slurry is then filtered on 0.7 µm filter, the extract pH is determined and the extract is analyzed for total metals (including Ag, Al, As, B, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Hg, K, Li, Mg, Mn, Mo, Na, Ni, P, Pb, Sb, Se, Si, Sn, Sr, Th, Ti, Tl, U, V, W, Y, and Zn).

Modified Acid Base Accounting

The modified Acid Base Accounting (ABA) test provided quantification of the total sulphur, sulphide sulphur, and sulphate concentrations present and the potential acid generation (AP) related to the oxidation of the sulphide sulphur concentration. The test method determined the neutralization potential (NP) of the samples by initiating a reaction with excess acid and then identified the quantity of acid neutralized by the samples, NP, by back-titrating to pH 8.3 with NaOH. The balance between the AP and NP assists in defining the potential of the sample to generate acid drainage. In addition, quantification of the carbonate mineral content permitted calculation of the theoretical carbonate NP.

Net Acid Generation Testing

The net acid generation (NAG) test facilitates a reaction between the sample and concentrated hydrogen peroxide in order to force complete oxidation and reaction of the acidity produced with the neutralizing minerals present within the sample. This test provided confirmation of data obtained in the ABA test. It also indicated the quantities of acid produced, and thus the amount of additional alkalinity, if any, needed to neutralize this acidity.

Decant Solution Analysis

The supernatants were decanted from the settled pulp slurries and analyzed to quantify contaminant concentrations that may report to surface or ground water systems in a tailings pond setting. The solutions were collected and submitted for chemical analysis of the following parameters: pH, alkalinity, acidity, conductivity, Eh, TDS, TSS, F^-, ammonia ($NH_3^+NH_4^+$), Cl^-, SO_4^{2-}, NO_2^-, NO_3^-, total thiosalts, and total and dissolved metals as per the previously noted suite of parameters plus total hardness. In addition, only the cyanide destruction (CND) decant solutions were analyzed for total cyanide, weak acid dissociable cyanide, cyanate, and thiocyanate.

In conclusion

1. Whole rock and elemental analyses determined that the flotation, CND, and combined tailings samples were predominantly comprised of silicate with minor amounts of aluminium, iron, potassium, and calcium, while the neutralization samples were predominantly calcium and iron with very little silica, due a predominant gypsum composition.
2. Analysis of the Duparquet TCLP leachates reported all Directive 019 parameters were at concentrations within the prescribed limits.
3. Modified ABA testing classified the Duparquet flotation and combined tailings samples NPAG due to a readily available carbonate source of alkalinity and insignificant sulphide content, while the CND tailings had sufficient sulphide content to indicate that acid generation would be expected.
4. NAG testing confirmed the NPAG designation provided by the ASA for the flotation and combined tailings samples. The NAG analysis also suggests that the CND samples would not generate PAG conditions.
5. The Duparquet flotation tailings decant solutions were all found to have concentrations of contaminants that were within the regulatory limits.
6. All CND decant solutions, with the exception of the RW-39 CND Decant and Dup-41 CND Decant, had total cyanide concentrations in excess of the regulatory limit (1 ppm).
7. Each of the CND decant solutions were found to have dissolved arsenic concentrations an order of magnitude above the regulatory limit, while elevated iron levels were due to suspended solids.
8. The neutralization decants showed elevated arsenic and iron values but only exceeded the regulatory limits as total values indicating the contaminants were associated with the suspended solids and would be expected to settle out of the water column over time.
9. The combined tailings decant solutions also showed elevated arsenic and iron concentrations in the total solution analyses, with arsenic generally being above the Directive 019 limits.

13.1.13 Recommendations

13.1.13.1 Variability Testing

The QP recommends the following tests:

- Additional variability hardness tests throughout the Project. Several samples from each zone should undergo SMC, rod mill and ball mill Bond tests;
- A variability locked-cycle testing program for the existing tailings area, including a mix of tailings and mineralized material.

13.1.13.2 Alternate Process

The metallurgical results using the POX process have been positive. However, the QP believes that the BIOX and Albion processes should be re-investigated. Although these processes have been tested and evaluated, it has been almost ten years since these tests were done and advancements in these technologies may change the metallurgical or financial outcomes.

14. MINERAL RESOURCE ESTIMATES

The mineral resource estimate update for the Project (the "2022 MRE") was prepared using all available information. The main objective was to update the results of InnovExplo's previous mineral resource estimate for the Project, dated June 26, 2013 (Poirier et al., 2014). The updated estimate includes new drill holes on the Beattie, Donchester and Central Duparquet properties.

The effective date of the 2022 MRE is September 12, 2022.

14.1 Methodology

The resource area has an E-W strike length of 4.5 km, a width of approximately 1 km, and a vertical extent of 1,050 m below surface.

The 2022 MRE was prepared using GEOVIA GEMS 6.8.2.2 ("GEMS") and GEOVIA Surpac 2021 ("Surpac") software. GEMS was used for updating the mineralized domains and the compositing. Surpac was used for the estimation, which consisted of 3D block modelling and the inverse distance square ("ID2") interpolation method. Statistical, capping and variography studies were completed using Snowden Supervisor v8.13 and Microsoft Excel software.

The main steps in the methodology were as follows:

- Review and validation of the database;
- Validation of the geological model and interpretation of the mineralized units;
- Validation of the drill hole intercepts database, compositing database and capping values for geostatistical analysis and variography;
- Validation of the block model and grade interpolation;
- Revision of the classification criteria and validation of the clipping areas for mineral resource classification;
- Assessment of resources with "reasonable prospects for economic extraction" and selection of appropriate cut-off grades and pit shell; and
- Generation of a mineral resource statement.

14.2 Drill hole and channel sample database

Fifty-five (55) new diamond drill holes have been added to the previously compiled and verified master database of Poirier et al. (2014):

- 22 DDH from Beattie
- 27 DDH from Donchester
- 6 DDH from Central Duparquet

Two holes drilled on the Beattie Project in 2014 and 2015 (BD-14-01 and BD-15-01) had undergone later additional sampling of the mineralized zones as seen by the inspection of the drill core, however the assay results were not available. Those holes were not included in the database.

The updated database contains 904 DDH and 892 channels with gold assay results and coded lithologies from the drill core logs and channel descriptions. It contains 173,831 sampled intervals taken from 270,119 m of drilled core and 2,371 analyses from 1,827 m of channels.

The 904 DDH cover the 4.5-km strike length of the Project at a reasonably regular drill spacing of 50 m. The 892 channels are centred mostly on two mineralized domains, South and RWRS, with the remaining channels unevenly distributed in the eastern part of the Beattie Project and on the Central Duparquet Project.

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

14.3 Geological Model

The 60 mineralized domains used during the 2014 MRE (Poirier et al., 2014) were validated and updated with the data from the 55 new holes.

The mineralized domain wireframes were created and updated by digitizing an interpretation onto sections spaced 25 m apart or 12.5 m in areas with higher drill hole density. The interpretation of the main mineralized domains extends up to 100 m past the last known occurrence of mineralization. However, if barren intervals were encountered, the mineralized zones were extended only to the mid-distance between the last known occurrence of mineralization and the barren hole. A 50-m extension around the zones was used for the secondary mineralized domains.

A dilution envelope was defined as the parts of the block model that are not included in any of the mineralized domain solids. The solid for the envelope contains “floating” gold intersects for which continuity has not yet been demonstrated or interpreted.

The mineralized domains of the Project can be separated into three groups based on their geometry:

1. A dominant E-W group with very steep to vertical dips;
2. A NE- to ENE-trending group with moderate to steep dips; and
3. An ESE-trending group with moderate to steep dips.

Figure 14-1 shows a 3D isometric view of all the mineralized domains, including Group 2 (yellow) and Group 3 (magenta). The geometry defined by these three groups can be observed on several outcrops on the Project (Figure 14-2 and Figure 14-3). Crosscutting relationships observed in the field suggest the contemporaneous development of these different groups. Such observations are compatible with the interpreted protracted structural evolution of the DPMFZ as presented in sections 7.1.2 and 7.3.4.

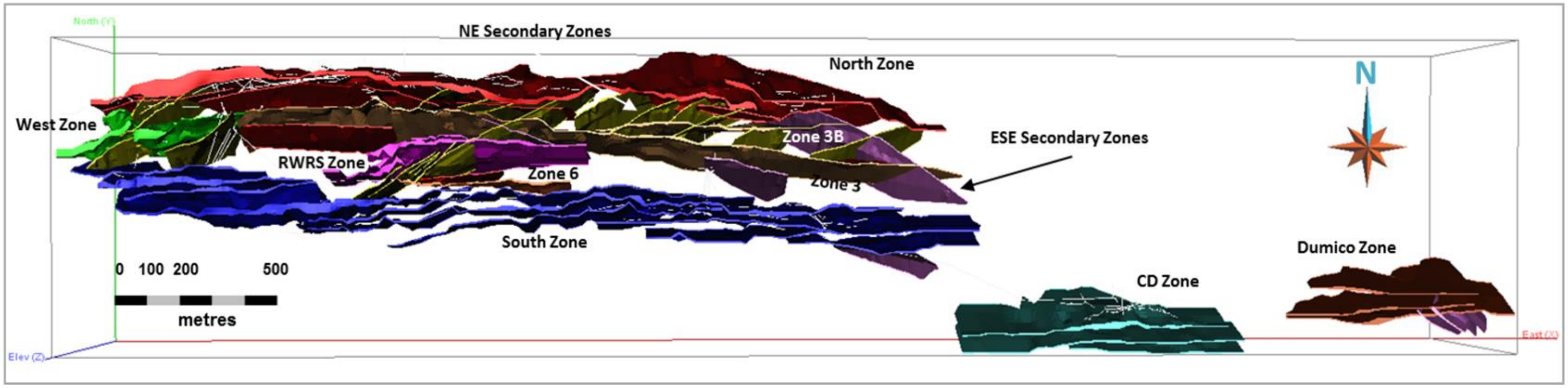
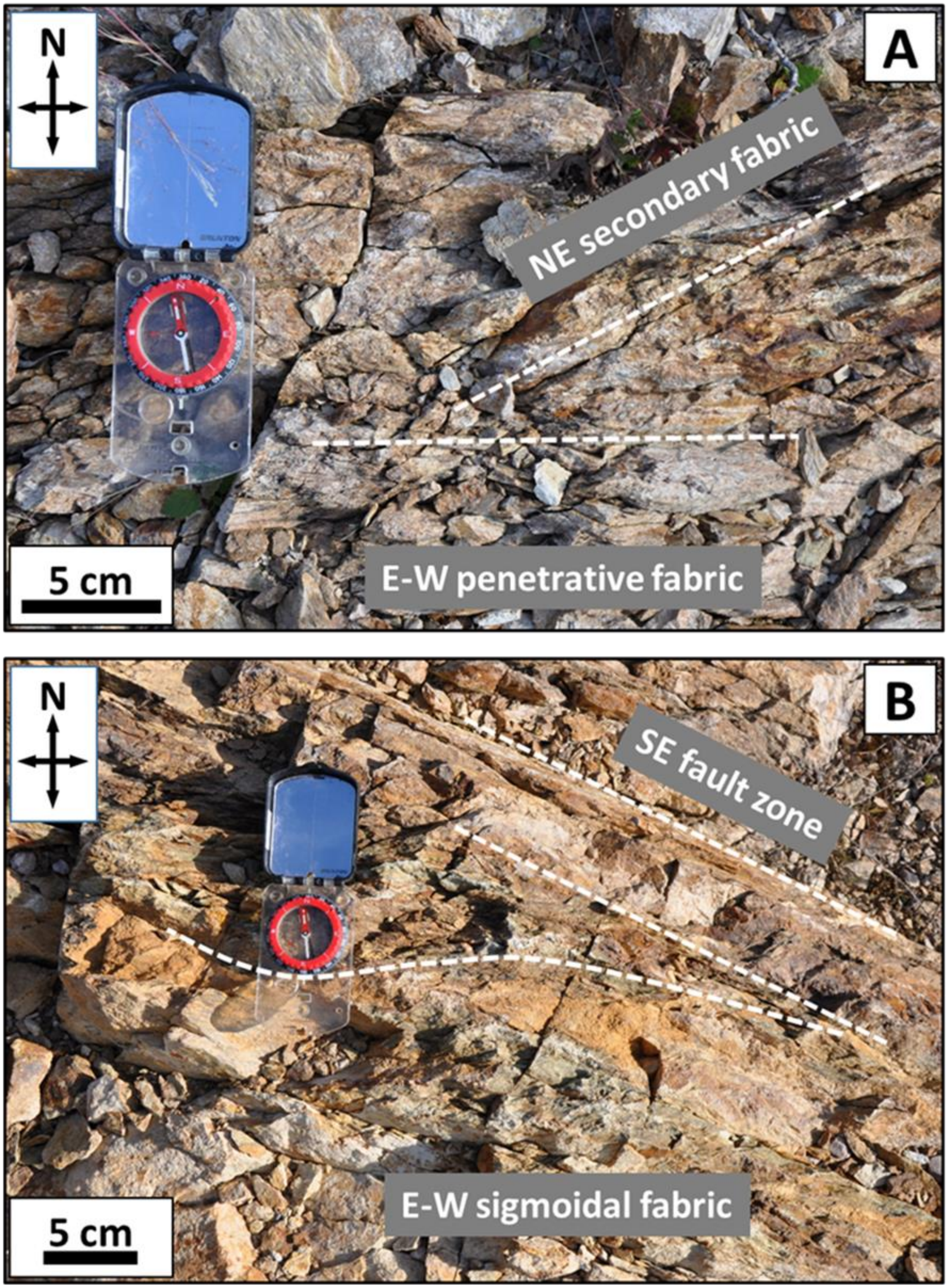
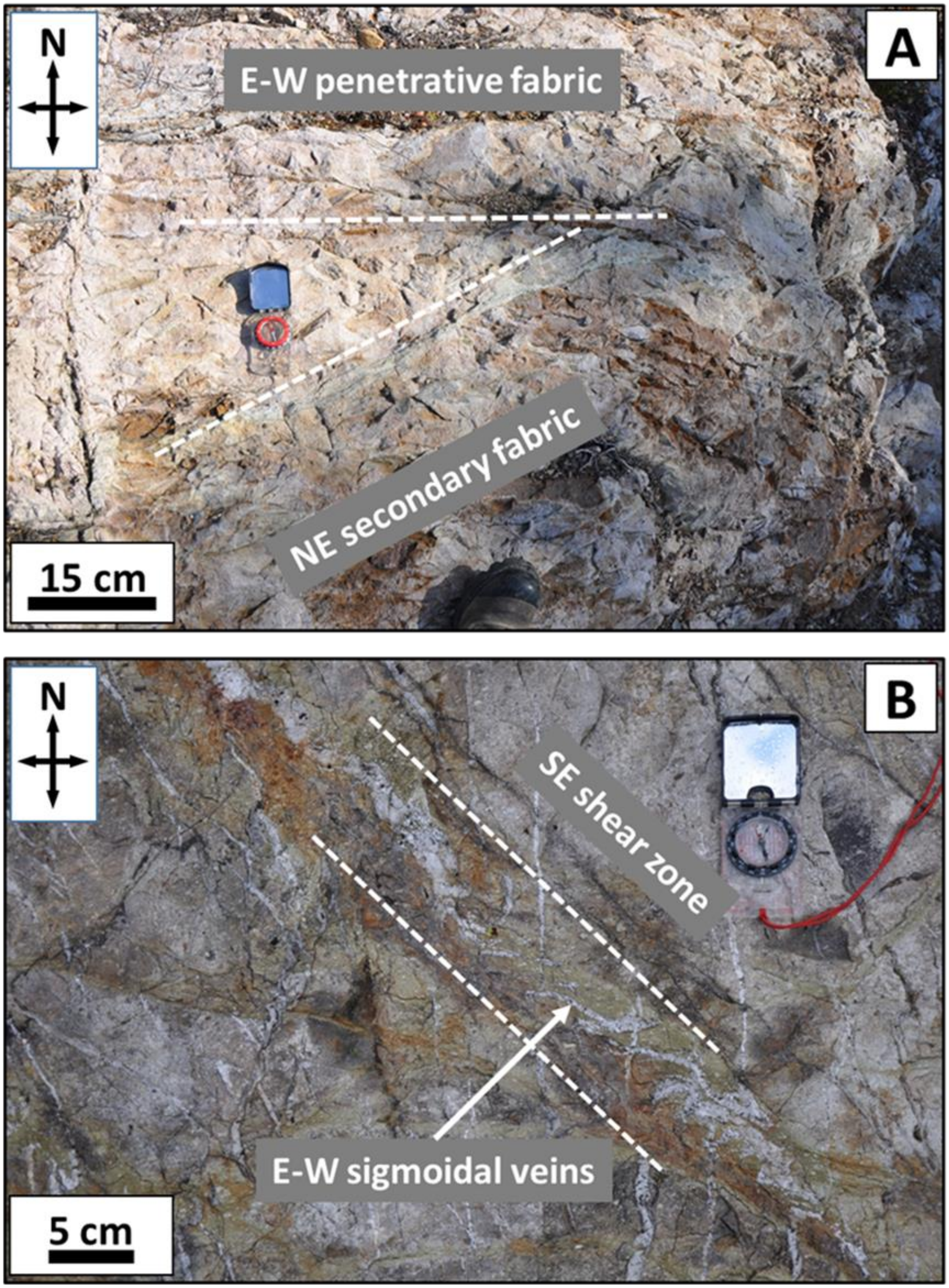


Figure 14-1 – General isometric view showing the main and secondary interpreted mineralized zones



A) Relationship between an E-W penetrative fabric and a NE-trending secondary fabric, subparallel to RWRS; B) Dominant SE-trending fault zone and E-W penetrative sigmoidal fabric.

Figure 14-2 – Outcrop photographs of structural features in the RWRS area



A) South Zone area: Relationship between an E-W penetrative fabric and a secondary NE-trending secondary fabric. B) Dumico area: A dominant SE-trending shear zone containing E-W trending sigmoidal quartz veins.

Figure 14-3 – Outcrop photographs of structural features in the South Zone and Dumico areas

14.4 High-grade Capping

Basic univariate statistics were performed on the raw assay datasets, grouped by zone. The following criteria were used to decide if capping was warranted:

- The coefficient of variation of the assay population is above 3.0.
- The quantity of metal contained in the top 10% highest grade samples is above 40%, and/or the quantity in the top 1% highest grade samples is higher than 10%.
- The probability plot of the grade distribution shows abnormal breaks or scattered points outside the main distribution curve.
- The log-normal distribution of grades shows erratic grade bins or distanced values from the main population.

The capping threshold decided for all domains is consistent with the combination of three criteria:

- A break in the probability plot.
- A coefficient of variation below 3.0 after capping.
- The total metal contained in the top 1% highest grade samples is below 10% after capping.

High-grade capping was set at 25 g/t Au for all zones, including the envelope zone. Twenty-two (22) DDH samples and one (1) channel sample were capped. Table 14-1 summarizes the statistical analysis by metal for domains with more than 100 samples. Figure 4-1 shows an example of graphs supporting the capping threshold decisions.

Table 14-1– Summary of univariate statistics on raw assays (domains with more than 100 samples)

Domain name	No. of samples	Uncut mean grade (g/t)	Uncut COV	Max grade (g/t)	High-grade capping (g/t)	No. of cut samples	Cut mean grade (g/t)	Cut COV	% of samples cut	% loss metal factor
NORTH	6,259	1.44	3.17	322.00	25	4	1.38	1.41	0.06%	4.94%
SOUTH_01	675	0.98	1.43	10.17	25		0.98	1.43		
SOUTH_02	2,135	1.14	2.08	87.80	25	1	1.11	1.39	0.05%	3.14%
SOUTH_03	1,232	0.81	1.53	26.90	25	1	0.81	0.50	0.08%	0.15%
SOUTH_04	2,241	0.89	1.40	11.52	25		0.89	1.40		
SOUTH_05	1,125	0.71	2.07	17.00	25		0.71	2.07		
SOUTH_06	1,743	0.87	2.56	37.58	25	1	0.87	2.45	0.06%	0.80%
SOUTH_07	378	1.25	2.50	34.90	25	1	1.23	2.35	0.26%	2.67%
SOUTH_08	348	0.95	2.65	33.12	25	1	0.92	2.42	0.29%	2.06%
SOUTH_09	166	0.86	1.89	8.07	25		0.86	1.89		
SOUTH_10	132	0.76	3.16	25.37	25	1	0.75	3.13	0.76%	0.34%
3	1,167	0.89	2.28	44.90	25	2	0.88	1.98	0.17%	1.58%
3B	878	0.99	2.63	59.59	25	2	0.95	1.97	0.23%	3.85%

Domain name	No. of samples	Uncut mean grade (g/t)	Uncut COV	Max grade (g/t)	High-grade capping (g/t)	No. of cut samples	Cut mean grade (g/t)	Cut COV	% of samples cut	% loss metal factor
WEST_1	1,361	1.46	2.50	119.50	25	2	1.39	1.31	0.15%	5.09%
WEST_2	721	1.88	1.44	37.67	25	1	1.88	1.35	0.14%	0.66%
WEST_3	220	1.39	0.81	5.82	25		1.39	0.81		
RWRS	2,699	1.30	1.58	24.01	25		1.30	1.58		
6	427	1.14	2.39	24.48	25		1.14	2.39		
CD	339	1.82	1.39	15.71	25		1.82	1.39		
CD_SOUTH	162	0.43	2.29	8.01	25		0.43	2.29		
O_01	215	1.20	1.76	16.15	25		1.20	1.76		
O_02	149	0.50	2.23	9.21	25		0.50	2.23		
O_07	161	0.52	2.13	7.24	25		0.52	2.13		
O_08	112	0.69	1.71	6.03	25		0.69	1.71		
O_09	275	0.51	1.64	8.05	25		0.51	1.64		
O_11	183	0.60	2.93	11.55	25		0.60	2.93		
O_12	127	0.97	1.82	12.87	25		0.97	1.82		
O_14	124	0.30	1.57	2.22	25		0.30	1.57		
O_15	232	0.47	1.91	5.62	25		0.47	1.91		
O_16	139	0.44	1.52	3.57	25		0.44	1.52		
O_17	248	0.77	1.92	10.20	25		0.77	1.92		
O_18	182	0.43	2.34	8.33	25		0.43	2.34		
O_19	121	0.29	2.19	3.53	25		0.29	2.19		
O_20	195	0.87	2.34	22.39	25		0.87	2.34		
O_25	119	0.97	2.21	16.03	25		0.97	2.21		
O_27	101	0.77	2.82	18.85	25		0.77	2.82		
O_28	117	0.42	1.75	4.31	25		0.42	1.75		
O_29	277	0.51	2.11	8.58	25		0.51	2.11		
O_30	109	0.62	2.41	11.87	25		0.62	2.41		
O_31	119	0.39	2.64	9.32	25		0.39	2.64		
O_33	181	0.27	2.40	5.60	25		0.27	2.40		
O_38	330	0.30	2.19	4.98	25		0.30	2.19		
O_46	171	0.74	1.54	8.27	25		0.74	1.54		
O_49	105	0.35	1.24	2.57	25		0.35	1.24		
O_50	123	0.76	4.58	37.30	25	1	0.66	3.69	0.83%	14.38%
DUM_03	124	1.36	2.58	27.89	25	1	1.33	2.50	0.82%	1.56%
Envelop	143,808	0.08	12.56	341.00	25	3	0.08	4.52	0.01%	4.00%

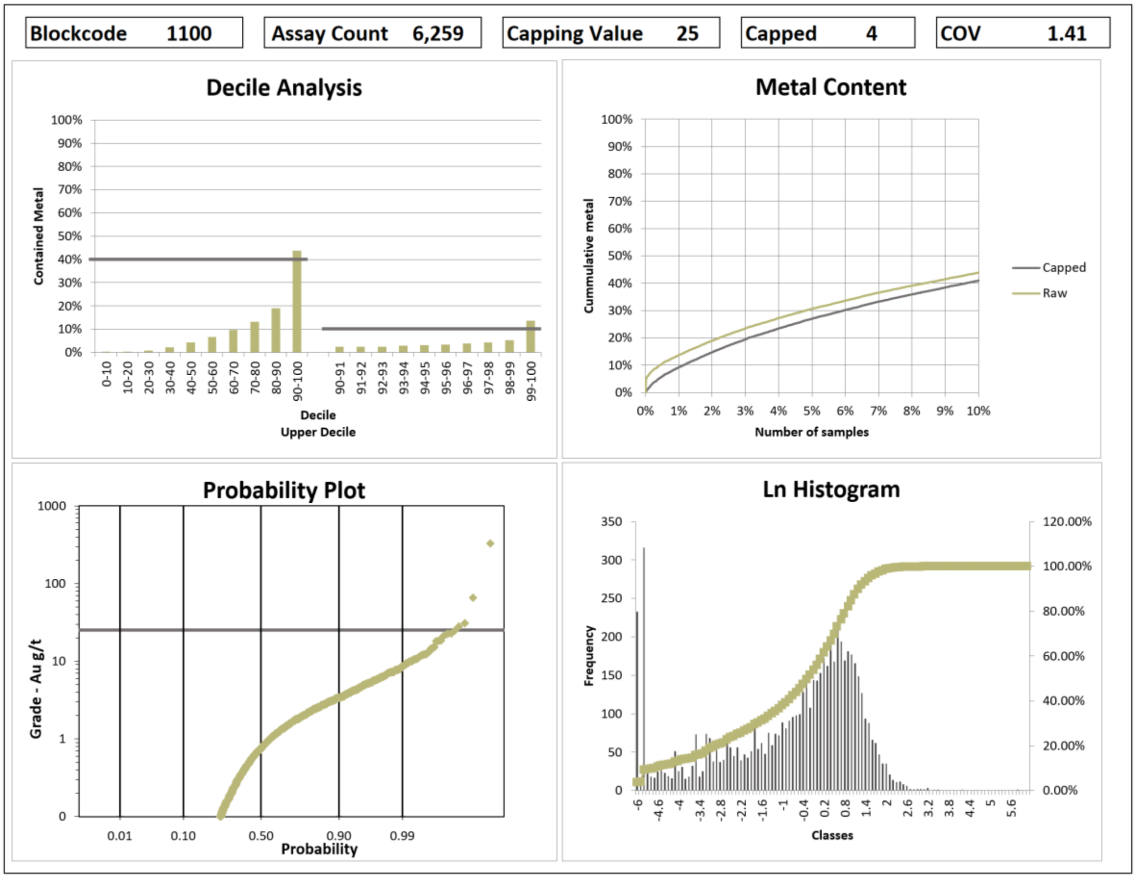


Figure 14-4 – Example of graphs supporting the capping decision for the North domain

14.5 Density

No new density data were collected on the 2013 to 2018 holes. Therefore, the current estimation relied on InnovExplo's previous analysis of 3,633 bulk density values and a 12 t composite drill core bulk sample from the mineralized zones, which yielded 2.73 g/cm^3 (mean density of the syenite) for all 60 mineralized domains and the envelope (Poirier et al., 2014). A density of 2.00 g/cm^3 was assigned to the overburden, and 1.00 g/cm^3 was assigned to excavation solids (drifts and stopes) assumed to be filled with water.

Density analyses were performed by ALS Canada in North Vancouver, BC and by Accurassay in Thunder Bay, Ontario on selected pulps from core samples. The determinations method employs an automated gas displacement pycnometer to determine density by measuring the pressure change of helium within a calibrated volume.

14.6 Compositing

The capped assays were composited within each mineralized domain to minimize any bias introduced by variations in sample lengths. The thickness of the mineralized domains, the proposed block size and the original sample length were considered when selecting the composite length.

The intervals defining each mineralized domain were composited to 1-m equal lengths. A grade of 0.00 g/t was assigned to missing sample intervals as it was assumed that unsampled intervals were considered to be unmineralized by the geologist in charge of the core logging. A total of 257,705 composites were generated from the DDH database, discarding tails less than 0.25 m long, and 2,298 composites were generated from the channel database. No composites were filtered out of the channel sample dataset due to the small amount of data and the fact that such filtering would be statistically meaningless.

Table 14-2 summarizes the basic statistics for the raw data and composites for the major domains (domains with more than 500 samples).

Table 14-2 – Summary statistics for the raw data and composites (major domains)

Domain name	No. of raw assays samples	Raw assays Max grade (g/t)	Raw assays Mean grade (g/t)	Raw assays COV	No. of comp.	Comp. Max grade (g/t)	Comp. Mean grade (g/t)	Comp. COV
NORTH	6,259	322.00	1.44	3.17	7,726	25.00	1.26	1.43
SOUTH_01	675	10.18	0.98	1.43	914	8.06	0.80	1.46
SOUTH_02	2,135	87.80	1.14	2.08	2,885	23.72	1.01	1.39
SOUTH_03	1,232	26.90	0.81	1.53	1,853	22.67	0.73	1.45
SOUTH_04	2,241	11.52	0.89	1.40	2,952	10.96	0.81	1.36
SOUTH_05	1,125	17.00	0.71	2.07	1,536	16.97	0.60	2.13
SOUTH_06	1,743	37.58	0.87	2.56	2,496	25.00	0.71	2.51
3	1,167	44.90	0.89	2.28	1,631	24.99	0.78	1.81
3B	878	59.59	0.99	2.63	1,201	25.00	0.84	2.09
WEST_1	1,361	119.50	1.46	2.50	1,840	25.00	1.39	1.27
WEST_2	721	37.67	1.88	1.44	909	20.95	1.82	1.29
RWRS	2,699	24.01	1.30	1.58	3,761	23.98	1.20	1.53
ENVELOPE	143,808	341.00	0.08	12.56	225,747	25.00	0.06	4.27

14.7 Block Model

A block model was built to enclose a sufficiently large enough volume to host an open pit. The model corresponds to a sub-blocked model in Surpac with no rotation. The user block size was defined as 5m x 5m x 5m with a minimal sub-block size of 1.25m x 1.25m x 1.25m. Block dimensions reflect the sizes of mineralized domains and plausible mining methods. All blocks with more than 50% of their volume falling within a selected solid were assigned the corresponding solid block code. Table 14-3 lists the properties of the block model. Table 14-4 provides details about the naming convention for the corresponding Surpac solids and the rock codes and precedence assigned to each solid.

Table 14-3 – Block model properties

Properties	Y (rows)	X (columns)	Z (levels)
Min. coordinates	5,373,300	628,500	-760
Max. coordinates	5,375,075	634,400	370
User block size	5	5	5
Min. block size	1.25	1.25	1.25
Rotation	0	0	0

Table 14-4 – Block model naming convention and rock codes

Domain Name	Description	Rock code	Precedence
Air	Air	5	1
Stope	Mined out	8	2
Drift Duparquet	Mined out	9	3
Drift CD	Mined out	9	3
OB	Overburden	6	6
NORTH	Mineralized domain	1100	10
SOUTH_01	Mineralized domain	1210	21
SOUTH_02	Mineralized domain	1220	22
SOUTH_03	Mineralized domain	1230	23
SOUTH_04	Mineralized domain	1240	24
SOUTH_05	Mineralized domain	1250	25
SOUTH_06	Mineralized domain	1260	26
SOUTH_07	Mineralized domain	1270	27
SOUTH_08	Mineralized domain	1280	28
SOUTH_09	Mineralized domain	1290	29
SOUTH_10	Mineralized domain	1295	21
3	Mineralized domain	1300	30
3B	Mineralized domain	1310	31
WEST_1	Mineralized domain	1410	41
WEST_2	Mineralized domain	1420	42
WEST_3	Mineralized domain	1430	43
RWRS	Mineralized domain	1500	50
6	Mineralized domain	1600	60
CD	Mineralized domain	1700	70
CD_SOUTH	Mineralized domain	1710	71
CD_S2	Mineralized domain	1720	72
O_01	Mineralized domain	2010	201

Domain Name	Description	Rock code	Precedence
O_02	Mineralized domain	2020	202
O_05	Mineralized domain	2050	205
O_06	Mineralized domain	2060	206
O_07	Mineralized domain	2070	207
O_08	Mineralized domain	2080	208
O_09	Mineralized domain	2090	209
O_10	Mineralized domain	2100	210
O_11	Mineralized domain	2110	211
O_12	Mineralized domain	2120	212
O_13	Mineralized domain	2130	213
O_14	Mineralized domain	2140	214
O_15	Mineralized domain	2150	215
O_16	Mineralized domain	2160	116
O_17	Mineralized domain	2170	217
O_18	Mineralized domain	2180	218
O_19	Mineralized domain	2190	219
O_20	Mineralized domain	2200	220
O_24	Mineralized domain	2240	224
O_25	Mineralized domain	2250	225
O_27	Mineralized domain	2270	227
O_28	Mineralized domain	2280	228
O_29	Mineralized domain	2290	229
O_30	Mineralized domain	2300	230
O_31	Mineralized domain	2310	231
O_33	Mineralized domain	2330	233
O_38	Mineralized domain	2380	238
O_40	Mineralized domain	2400	240
O_41	Mineralized domain	2410	241
O_42	Mineralized domain	2420	242
O_46	Mineralized domain	2460	146
O_48	Mineralized domain	2480	248
O_49	Mineralized domain	2490	249
O_50	Mineralized domain	2500	250
DUM_01	Mineralized domain	3110	311
DUM_02	Mineralized domain	3120	312
DUM_03	Mineralized domain	3130	313
DUM_04	Mineralized domain	3510	314

Domain Name	Description	Rock code	Precedence
DUM_05	Mineralized domain	3520	315
ENVELOP	Envelop	4000	400

14.8 Variography and Search Ellipsoids

Individual zones and the envelope were estimated separately using their own search ellipsoid. The available geological and geostatistical information was used to establish the size of each search ellipsoid.

Eight (8) of the mineralized zones were divided into panels based on variations in internal geometry. The criteria used for determining whether a zone should be subdivided were the intensity of the along-strike or down-dip curvature and/or the presence of "splays" associated with the given zone.

Table 14-5 lists the zone's panels, their block codes and mean orientations. Figure 14-7 illustrates the panel subdivisions of the North Zone and the cutting planes used.

Table 14-5 – Mean orientation of the panels for the geometrically irregular mineralized domains

Domain	Panel	Block code	Azimuth (°)	Dip (°)
NORTH	1	1101	80	80
	2	1102	275	80
	3	1103	265	80
	4	1104	90	70
SOUTH_1	1	1211	275	85
	2	1212	95	75
SOUTH_2	1	1221	275	85
	2	1222	85	87.5
SOUTH_3	1	1231	90	70
	2	1232	280	80
SOUTH_4	1	1241	85	87.5
	2	1242	280	70
SOUTH_8	1	1281	265	90
	2	1282	275	85
SOUTH_10	1	1296	275	85
	2	1297	265	80
RWRS	1	1501	90	80
	2	1502	90	75
	3	1503	65	75
	4	1504	85	85

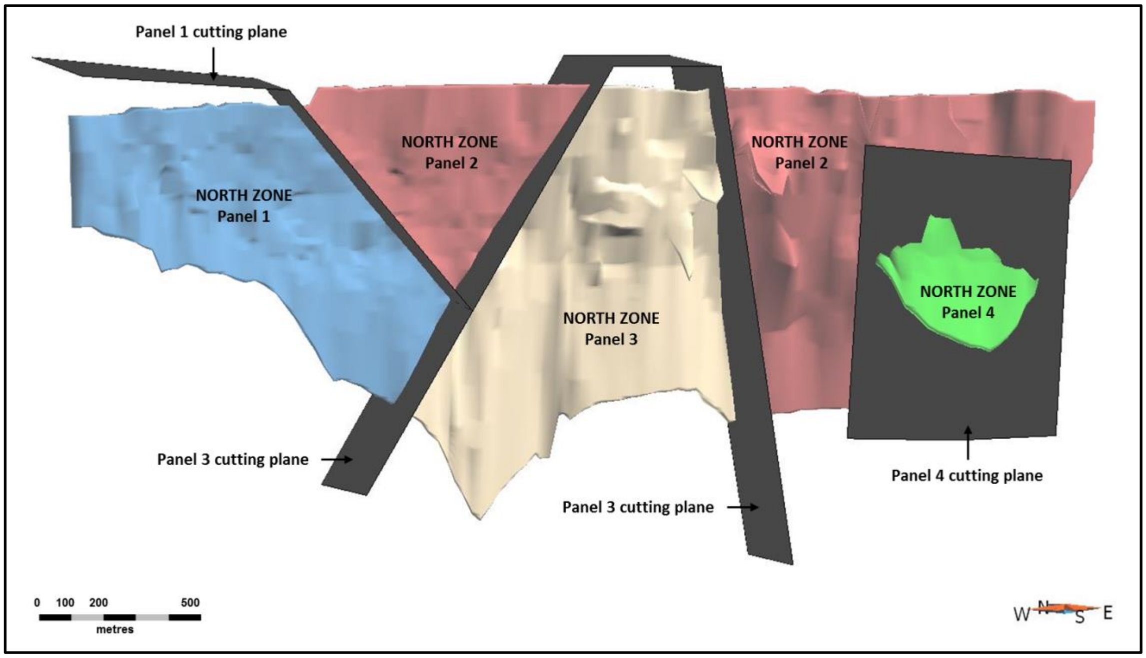


Figure 14-5 – NNE vertical section showing the panel subdivisions of the North Zone

Seventy-two (72) distinct mineralized domains were studied. The domains were based on the panels discussed above. The mean azimuth and dip of each domain were used to define 23 search ellipsoids, grouping zones of similar geometry. The best result for the envelope zone was obtained using an orientation of N090/90, which corresponds to the average attitude of the structural trend on the Project.

Ellipsoid radiuses were established using a combination of the ranges determined from the geostatistical analysis of the main zones and the interpretation of the true thickness of each zone. The radius of the third axis (Z) varies from 15 to 30 m, reflecting the density data for each zone and the confidence level of the geological interpretation.

Table 14-6 summarizes the parameters of the ellipsoids used for interpolation.

Table 14-6 – Search ellipsoid parameters

Ellipsoid	Rotation *			Radius			Orientation		Domain Rock code	Panel
	Z (°)	X (°)	Z (°)	X (m)	Y (m)	Z (m)	Azimuth (°)	Dip (°)		
1	50	75	0	80	80	15	40	75	2050	
2	40	70	0	80	80	15	50	70	2280	
3	40	87	0	80	80	15	50	87	2120	
									2080	
									2100	
									2110	
									2070	
									2170	
4	30	75	0	80	80	15	60	75	2420	
									2240	
									2060	
									2310	
									2190	
									2250	
									2300	
									2150	
									1500	1503
5	30	85	0	80	80	15	60	85	2490	
									2500	
6A	15	80	0	80	80	15	65	80	1410	
6B	20	80	0	80	80	15	70	80	2140	
							70	80	2130	
7	15	40	0	80	80	15	75	40	1420	
8	15	85	0	80	80	15	75	85	1400	
9	10	75	0	110	110	15	80	75	2020	
				110	110	25	80	80	1100	1101
10	5	85	0	110	110	15	80	85	2010	
				110	110	25	85	85	1500	1504
				110	110	25	85	85	1240	1241
				110	110	25	85	85	1270	
				110	110	25	85	85	1250	
				110	110	25	85	85	1260	
				110	110	25	85	85	1220	1222

<table>
<tr><th rowspan="2">Ellipsoid</th><th colspan="3">Rotation *</th><th colspan="3">Radius</th><th colspan="2">Orientation</th><th rowspan="2">Domain Rock code</th><th rowspan="2">Panel</th></tr>
<tr><th>Z (°)</th><th>X (°)</th><th>Z (°)</th><th>X (m)</th><th>Y (m)</th><th>Z (m)</th><th>Azimuth (°)</th><th>Dip (°)</th></tr>
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<tr><td>12A1</td><td>0</td><td>80</td><td>0</td><td>110</td><td>110</td><td>25</td><td>90</td><td>80</td><td>1500</td><td>1501</td></tr>
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<tr><td rowspan="4">12B</td><td rowspan="4">0</td><td rowspan="4">85</td><td rowspan="4">0</td><td>110</td><td>110</td><td>15</td><td>90</td><td>85</td><td>1600</td><td></td></tr>
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<tr><td rowspan="4">15</td><td rowspan="4">5</td><td rowspan="4">-85</td><td rowspan="4">0</td><td>110</td><td>110</td><td>25</td><td>265</td><td>80</td><td>1100</td><td>1103</td></tr>
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<tr><td>110</td><td>110</td><td>30</td><td>270</td><td>75</td><td>1710</td><td></td></tr>
<tr><td>110</td><td>110</td><td>30</td><td>270</td><td>75</td><td>1720</td><td></td></tr>
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<tr><td>110</td><td>110</td><td>15</td><td>275</td><td>85</td><td>2090</td><td></td></tr>
</table>

Ellipsoid	Rotation *			Radius			Orientation		Domain Rock code	Panel
	Z (°)	X (°)	Z (°)	X (m)	Y (m)	Z (m)	Azimuth (°)	Dip (°)		
				110	110	15	275	85	2410	
18	-10	-75	0	110	110	25	280	70	1240	1242
				110	110	25	280	80	1230	1232
19	-45	-70	0	110	110	30	300	65	3510	
				110	110	30	300	65	3520	
20	0	-90	0	50	50	15			4000	
**Block Model system: positive rotation is counter-clockwise.*										

14.9 Grade Interpolation

The variography study provided the parameters for interpolating the grade model using capped composites. The interpolation was run in Surpac on point area workspaces extracted from the composite datasets (flagged by domain). The interpolation profiles were applied to each mineralized domain using hard boundaries.

The method retained for the resource estimation was inverse distance square (“ID2”).

The two strategies and the parameters for the grade estimation are summarized in Table 14-7.

Table 14-7 – Interpolation strategies

Folder	Search ellipsoid range	Number of composites		
		Min	Max	Max per hole
Mineralized domains	X 1	2	12	4
Envelope	X 1	2	6	0

14.10 Block Model Validation

Block model grades and composite grades were visually compared on sections, plans and longitudinal views for densely and sparsely drilled areas. The grade distribution had a good match without excessive smoothing in the block model. The process confirmed that the block model honours the drill hole composite data (Figure 14-6).

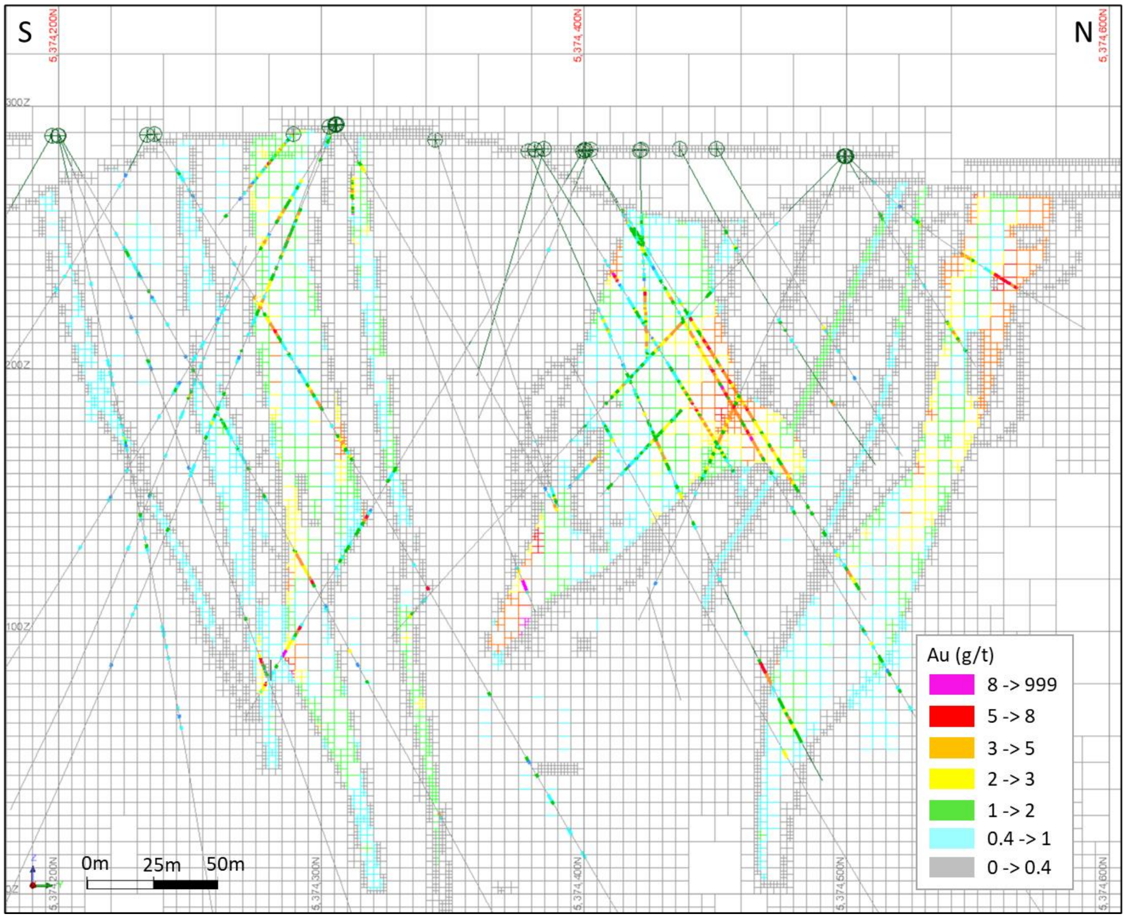


Figure 14-6 – block model interpolated gold values versus drill holes gold assays (section view 630,100E)

The trend and local variation of the estimated ID2 model were compared to the composite data using statistics and swath plots in three directions (North, East and Elevation). Table 14-8 compares the global block model mean and the composite grades for three (3) domains. Generally, the comparison between composite and block grade distribution did not identify any significant issues. Figure 14-7 shows an example of the swath plot used to compare the block model grades to the composite grades. In general, the model correctly reflects the trends demonstrated by the composites, with the expected smoothing effect.

Table 14-8 – Comparison of block model and composite mean grades for three domains

Domain Rock code	Project	Composite	ID2 Model	Difference
1100	Number	7828	2489635	
	Mean (g/t)	1.24	1.20	96%
	COV	1.44	0.97	68%
1300	Number	1631	885681	
	Mean (g/t)	0.78	1.81	97%
	COV	0.76	0.89	49%
1600	Number	527	92578	
	Mean (g/t)	1.03	2.31	96%
	COV	0.99	1.16	50%

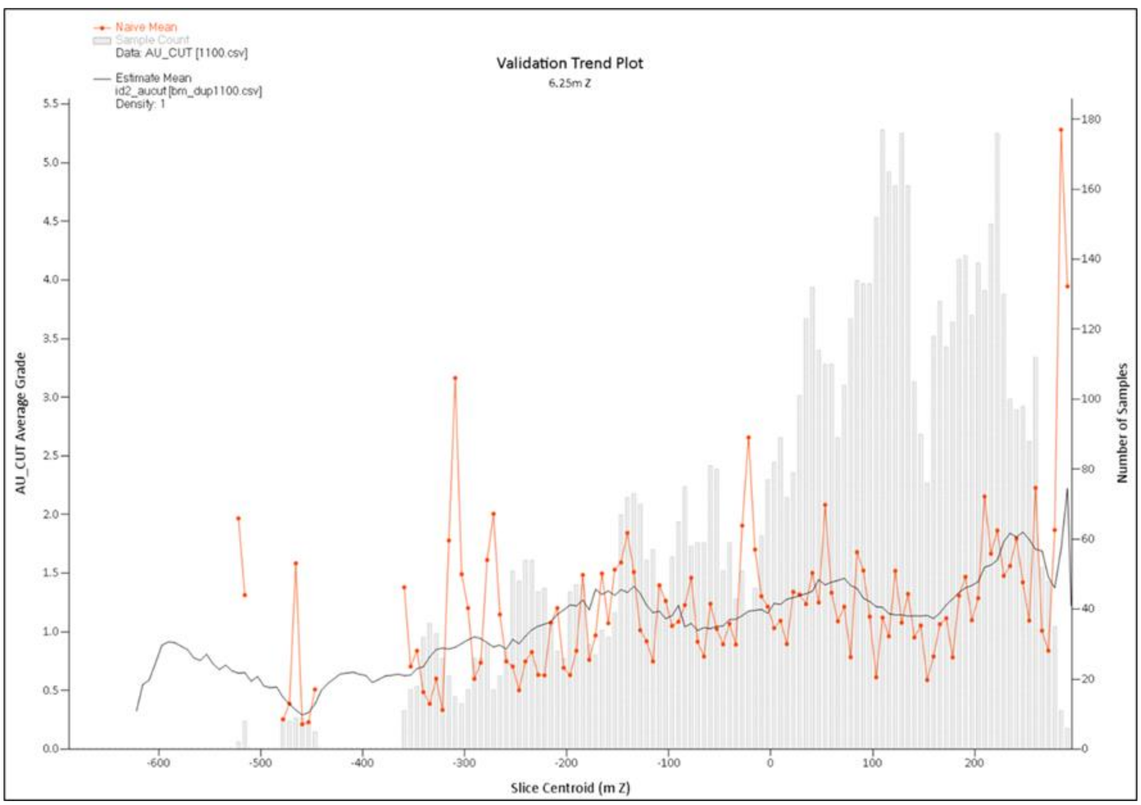


Figure 14-7 – Swath plot comparing the ID2 interpolation to the DDH composites for the 1100 domain (sliced by elevation)

14.11 Mineral Resource Classification

Measured resources were defined for blocks having a volume of at least 25% within an envelope built at a distance of 10 m around existing channel samples.

Indicated resources were defined for blocks meeting at least one (1) of the conditions below:

- Blocks falling within a 15 m wide buffer surrounding existing stopes
- Blocks for which the average distance to composites is less than 45 m

A clipping polygon was generated for each of the 60 domains using the criteria described above, and the blocks were coded accordingly. Only blocks for which reasonable geological and grade continuity have been demonstrated were selected. In some cases, isolated blocks were upgraded or downgraded to homogenize the model. All remaining interpolated blocks were classified as inferred resources. Figure 14-8 shows an example of the clipping polygon used to delimit the indicated resource for the North domain.

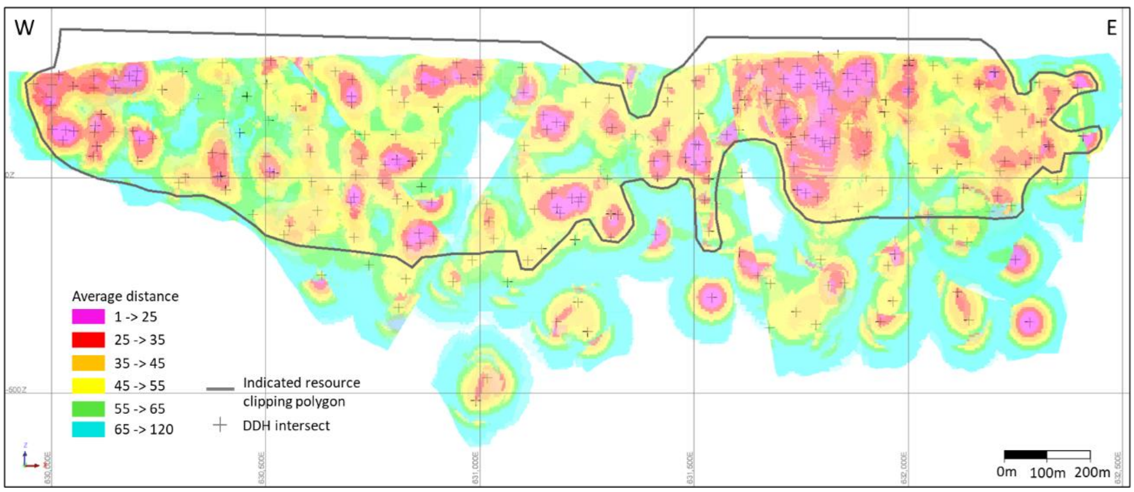


Figure 14-8 – Longitudinal view of the indicated resource clipping polygon used for the North domain (looking north)

Blocks interpolated in the envelope were all classified as inferred resources.

14.12 Cut-off Grade for Mineral Resources

Under CIM Definition Standards, mineral resources should have “reasonable prospects of eventual economic extraction”.

A Whittle pit shell was used to constrain the 2022 MRE for its near-surface potential. Resource-level optimized pit shells and the corresponding open-pit cut-off grade are used for the open pit resource statement. The remaining (out-pit) mineralized material was then flagged for its underground potential. Deswik Stope Optimizer (“DSO”) was used to apply constraining volumes to any blocks in the potential underground extraction scenario to address the reasonable prospect for eventual economic extraction of underground resources. An isometric and plan view (Figure 14-9) showing the optimized pit-shell and DSO stope designs of the classified mineral resources are provided to visualize relationships between the two.

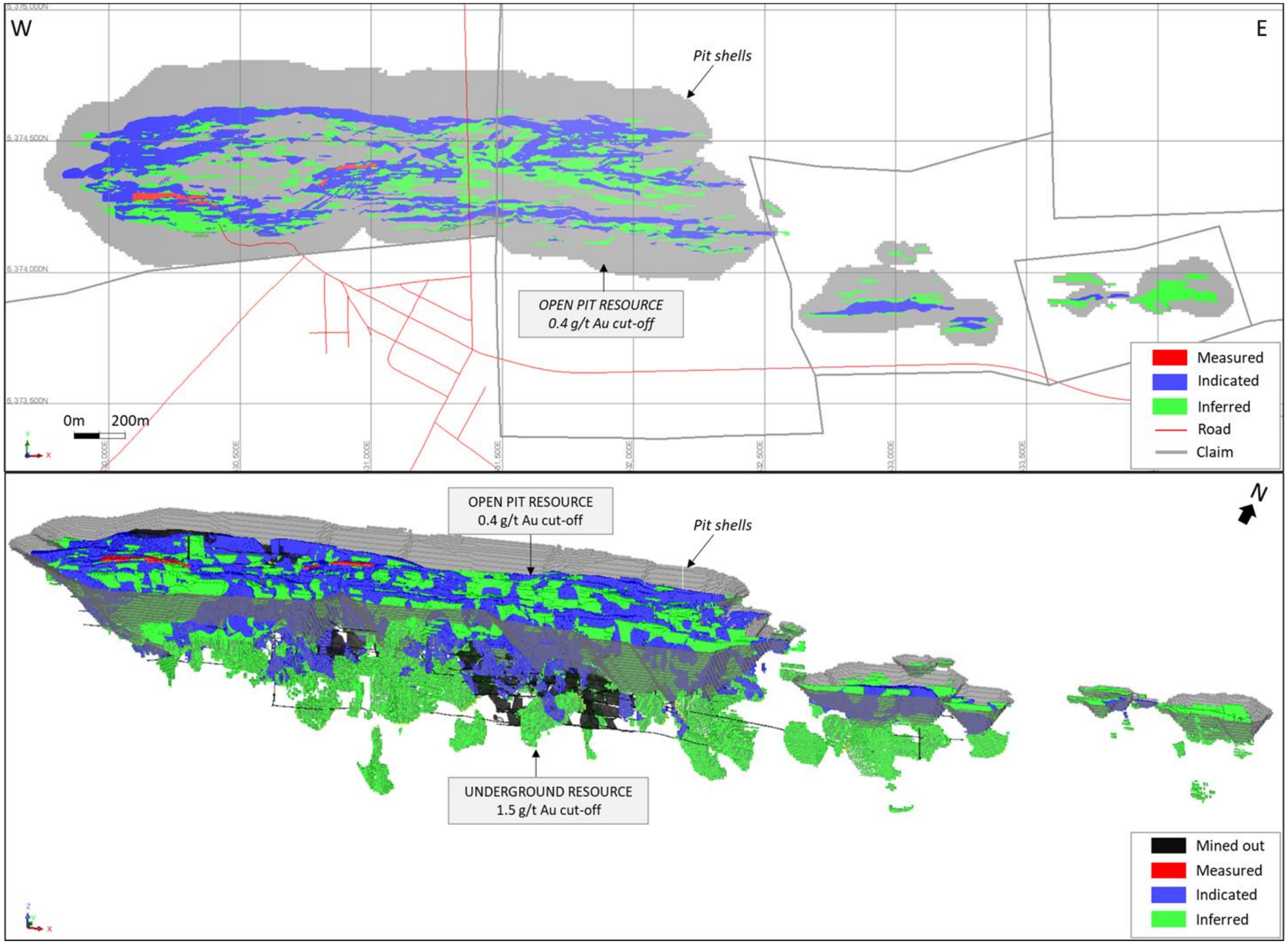


Figure 14-9 – Plan view and isometric view of the categorized mineral resources and the Whittle optimized pit-shells (blocks selection: in pit-shells or in DSO and above the respective COG)

Mineral resources were compiled using a minimum cut-off grade ("CoG") for two combined potential extraction scenarios: 0.4 g/t Au for open pit and 1.5 g/t Au for underground mining.

The CoG parameters and assumptions are presented in Table 14-9.

Table 14-9 – Input parameters used to estimate the cut-off grade

Parameter	Unit	Value for open pit	Value for underground mining
Gold price	US$/oz	1,650	1,650
Exchange rate	US:CA	1.31	1.31
Royalty	%	0	0
Refining cost	US$/oz	5.00	5.00
Cost of selling	US$/oz	5.00	5.00
Total processing cost	CA$/t treated	11.9	17.00
Metallurgical recovery	%	93.9	93.9
Mining method			Long hole
Mining cost	CA$/t treated		70.00
G&A	CA$/t treated	8.75	8.75
Total ore based cost	CA$/t treated	22.90	95.75
Cut-off grade	Au g/t	0.4	1.50

14.13 Mineral Resource Estimates

14.13.1 In-Pit and Underground Resource Estimates

The authors have classified the current mineral resource estimate as measured, indicated and inferred resources based on data density, search ellipse criteria, drill hole spacing and interpolation parameters. The authors also believe that the requirement of "reasonable prospects for eventual economic extraction" has been met by having:

- Resources constrained by a pit shell, with a 50° angle in rock and a 30° angle in overburden;
- Constraining volumes applied to any blocks (potential underground extraction scenario) using the Deswik Stope Optimizer (DSO) for the out-pit resources; and
- Cut-off grades based on reasonable inputs amenable to potential open-pit and underground extraction scenarios.

The 2022 MRE is considered reliable and based on quality data and geological knowledge. The estimate follows CIM Definition Standards.

Table 14-10 presents the results of the in-pit and underground portions of the 2022 MRE, combining potential open pit and underground mining scenarios at respective cut-off grades of 0.4 g/t Au and 1.5 g/t Au.

Table 14-11 presents the sensitivity of the in-pit and underground resources at different cut-off grades for each mining method. The reader should be cautioned that the figures provided in the sensitivity table should not be interpreted as a mineral resource statement. The reported quantities and grade estimates at different cut-off grades are presented for the sole purpose of demonstrating the sensitivity of the resource model to the selection of a reporting cut-off grade.

The in-pit and underground portion of the 2022 MRE represents an increase of 11% in the Indicated Resource ounces and an increase of 13% in the inferred Resource ounces compared to the previous 2014 MRE (Poirier et al., 2014). Measured Resource ounces decreased by 6.3% mainly due to the use of constraining volumes. This increase is due to the addition of the 55 new assayed holes that were drilled on the Project since 2013 and the adjustment of the economic parameters to reflect current economic conditions.

INNOVEXPLO

Table 14-10 – In-pit and underground portions of the Duparquet Project 2022 Mineral Resource Estimate by mining method

Area (mining method)	Cut-off (g/t)	Measured resource			Indicated resource			Inferred resource		
		Tonnage (t)	Au (g/t)	Ounces	Tonnage (t)	Au (g/t)	Ounces	Tonnage (t)	Au (g/t)	Ounces
Open pit	0.4	163,700	1.37	7,200	59,410,600	1.52	2,909,600	28,333,000	1.07	970,400
UG mining	1.5	-	-	-	5,506,900	2.26	399,300	9,038,900	2.29	665,600
Total	-	163,700	1.36	7,200	64,917,474	1.59	3,308,880	37,371,851	1.36	1,636,044

Notes to accompany the Mineral Resource Estimate:

1. The independent and qualified persons for the mineral resource estimate, as defined by NI 43-101are Marina Iund, P.Geo., Carl Pelletier, P.Geo., Simon Boudreau, P.Eng., all from Innovexplo and Guy Comeau, P.Eng. from Soutex. The effective date of the estimate is September 12, 2022.
2. These mineral resources are not mineral reserves, as they do not have demonstrated economic viability. There is currently insufficient data to define these Inferred mineral resources as Indicated or Measured mineral resources and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured mineral resource category. The mineral resource estimate follows current CIM Definition Standards.
3. The results are presented in situ and undiluted and have reasonable prospects of economic viability.
4. The estimate encompasses sixty (60) mineralized domains, and one (1) dilution envelope using the grade of the adjacent material when assayed or a value of zero when not assayed.
5. High-grade capping of 25 g/t Au supported by statistical analysis was done on raw assay data before compositing.
6. The estimate was completed using a sub-block model built in GEOVIA SURPAC 2021, a block size of 5m x 5m x 5m and a minimum block size of 1.25m x 1.25m x1.25m. Grades interpolation was obtained by ID2 using hard boundaries.
7. A density value of 2.73 g/cm^3 was used for the mineralized domains and the envelope. A density value of 2.00 g/cm^3 was used for the overburden. A density value of 1.00 g/cm^3 was used for the excavation solids (drifts and stopes) assumed to be filled with water.
8. The mineral resource estimate is classified as measured, indicated and inferred. The Measured category is defined by blocks having a volume of at least 25% within an envelope built at a distance of 10 m around existing channel samples. The Indicated category is defined by blocks meeting at least one (1) of the following conditions: Blocks falling within a 15-m buffer surrounding existing stopes and/or blocks for which the average distance to composites is less than 45 m. A clipping polygon was generated to constrain indicated resources for each of the 60 mineralized domains. Only the blocks for which reasonable geological and grade continuity have been demonstrated were selected. All remaining interpolated blocks were classified as Inferred resources. Blocks interpolated in the envelope were all classified as inferred resources.
9. The mineral resource estimate is locally pit-constrained with a bedrock slope angle of 50° and an overburden slope angle of 30°. The out-pit mineral resource met the reasonable prospect for eventual economic extraction by having constraining volumes applied to any blocks (potential underground extraction scenario) using DSO. It is reported at a rounded cut-off grade of 0.4 g/t Au (in-pit) and 1.5 g/t Au (UG). The cut-off grades were calculated using the following parameters: mining cost = CA$70.00 (UG); processing cost = CA$11.9 to 17.0; G&A = CA$8.75; refining and selling costs = CA$ 5.00; gold price = US$1,650/oz; USD:CAD exchange rate = 1.31; and mill recovery = 93.9%. The cut-off grades should be re-evaluated in light of future prevailing market conditions (metal prices, exchange rates, mining costs etc.).
10. The number of metric tons and ounces was rounded to the nearest hundred, following the recommendations in NI 43-101, and any discrepancies in the totals are due to rounding effects.
11. The authors are not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, or marketing issues, or any other relevant issue not reported in the Technical Report, that could materially affect the Mineral Resource Estimate.

Table 14-11 – Cut-off grade sensitivity for the in-pit and underground portions of the Duparquet Project

Area (mining method)	Cut-off (g/t)	Measured resource			Indicated resource			Inferred resource		
		Tonnage (Mt)	Au (g/t)	Ounces	Tonnage (Mt)	Au (g/t)	Ounces	Tonnage (Mt)	Au (g/t)	Ounces
Open pit	0.5	156,938	1.41	7,122	41,152,335	1.70	2,253,068	11,007,061	1.13	400,881
	0.45	161,081	1.39	7,187	53,548,726	1.58	2,722,586	22,032,449	1.16	824,601
	0.4	163,709	1.37	7,222	59,410,612	1.52	2,909,551	28,332,980	1.07	970,424
	0.35	165,800	1.36	7,248	66,307,600	1.46	3,117,172	37,354,222	0.96	1,147,282
UG mining	1.9	-	-	-	5,891,904	2.67	505,871	7,168,869	2.91	669,750
	1.7	-	-	-	5,224,787	2.47	414,153	7,378,504	2.51	595,956
	1.5	-	-	-	5,506,861	2.26	399,356	9,038,871	2.29	665,629
	1.3	-	-	-	5,302,381	2.10	357,603	11,459,118	2.05	756,440

14.13.2 Tailings Resource Estimate

In August 2009, Genivar was contracted by Clifton Star to produce a NI 43-101 compliant resource estimate for the Beattie mine tailings (Fillion, 2009), but the report was never published (Figure 14-10). On November 7, 2011, Clifton Star mandated InnovExplo to audit the 2009 results to incorporate them into a consolidated mineral resource estimate, which was published in an NI 43-101 technical report on July 5, 2012 (Brousseau et al., 2012).

For the current report, the authors reviewed the parameters of both MREs. The only change from 2009 to 2012 was that the volumetrics were re-calculated using a cut-off grade similar to the cut-off grade defined for the in-pit component of the current 2022 MRE (see Section 14.13.1).

Genivar's 2009 MRE was based on fieldwork performed in May, June and September 2009. During the fieldwork:

- 271 stations were surveyed;
- 449 samples were assayed for gold and specific gravity measurements; and
- 24 samples were collected for bulk density measurements.

Figure 14-11 shows the location of the different zones in the tailings and sample distribution.

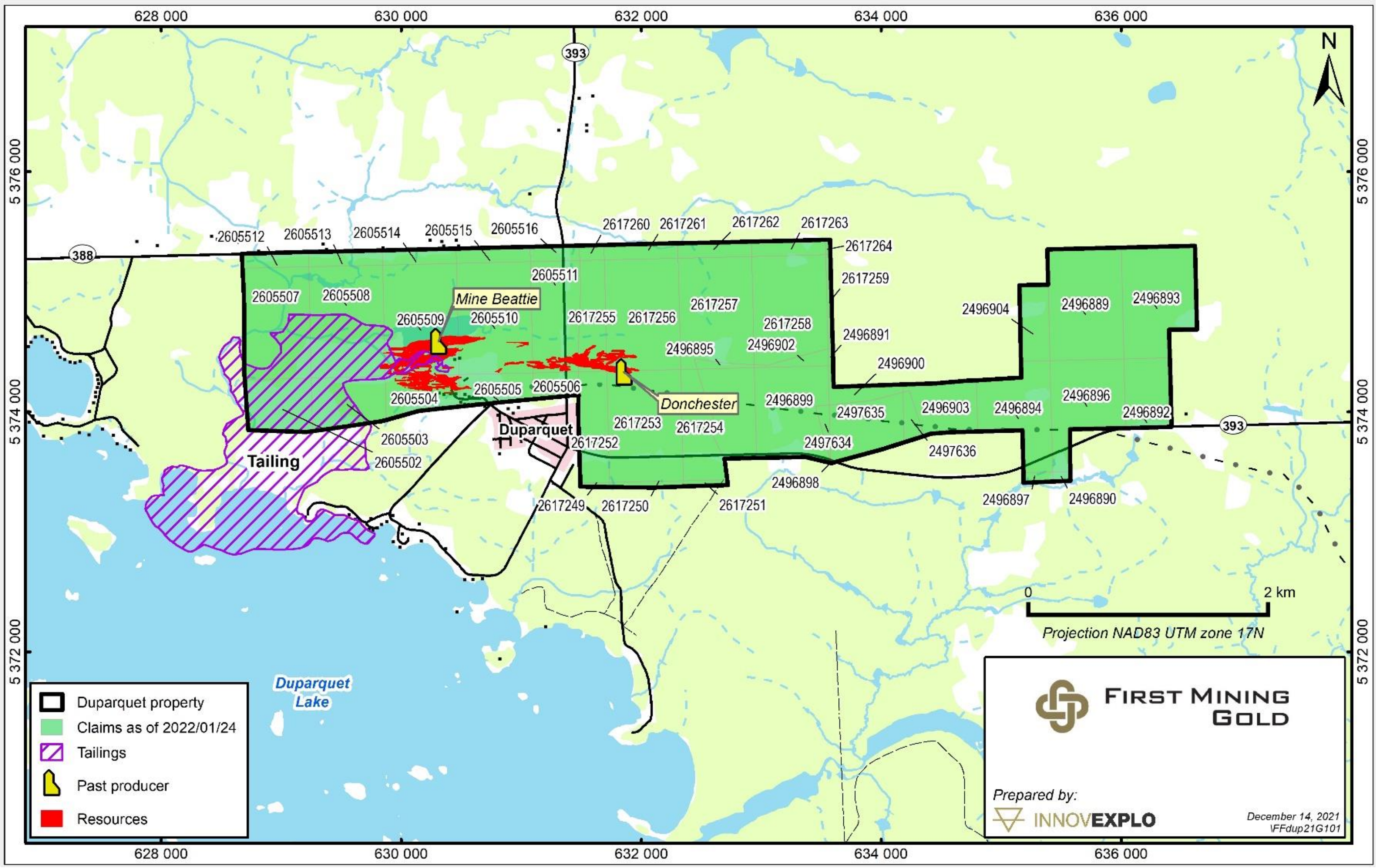
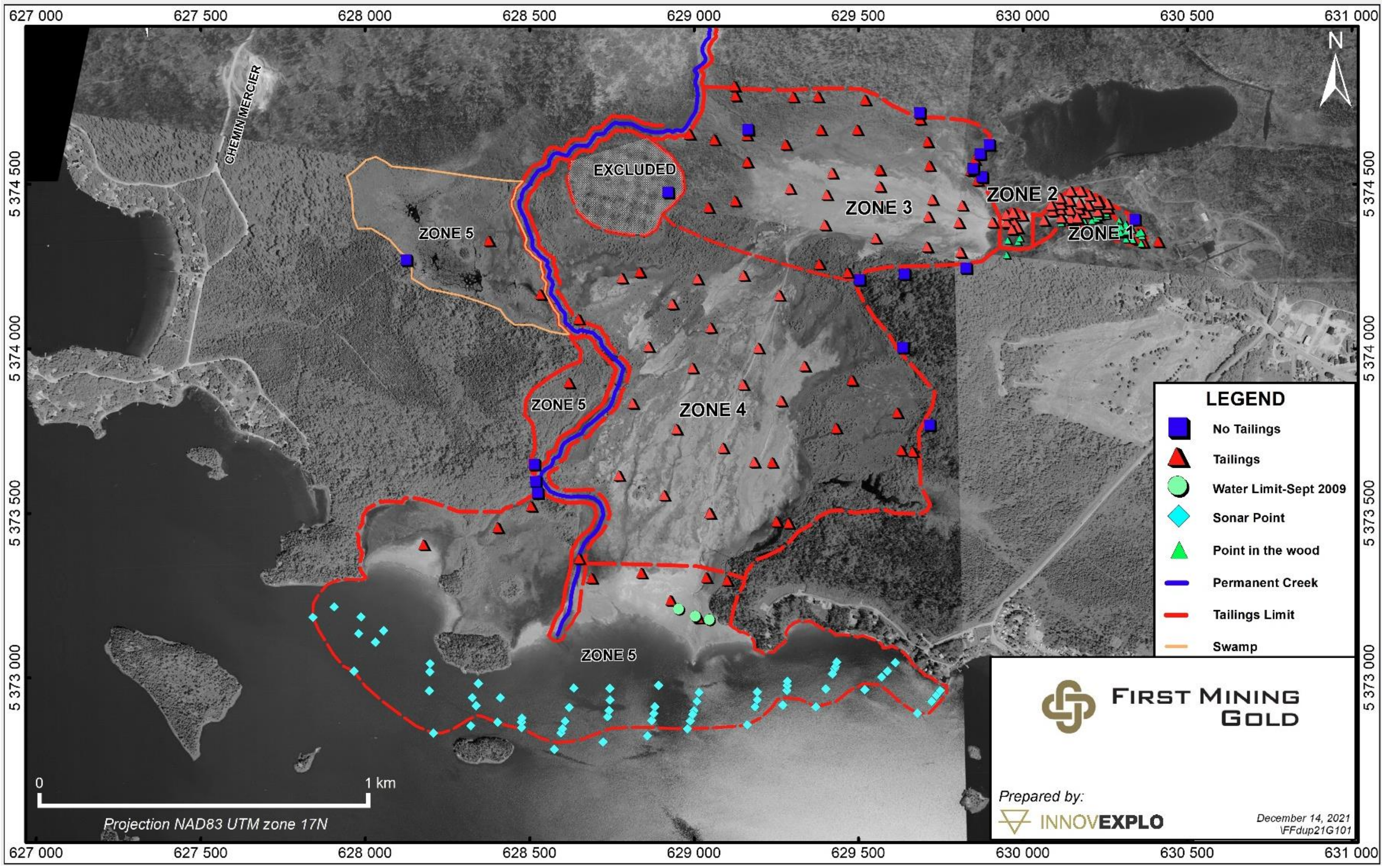


Figure 14-10 – Location map showing the mining titles comprising the Duparquet Project and the location of the tailings resource



From Genivar, 2009

Figure 14-11 –Tailings coverage and location of sampling points

The boundaries of the tailings were surveyed with a high-precision GPS (Mobile GPS – Leica, System 1200), which provided the topography. The holes were drilled down to the bottom contact with the clays, "gumbo" (basal clay) and organic material.

The block model was built using GEOVIA GEMS software. The dimensions of the blocks were 5m x 5m x 1m. The intervals defining each favourable lithology were composited to equal lengths of 0.5 m. The high values were capped as follows:

- Zone 1: 13.0 g/t Au
- Zone 2: 3.5 g/t Au
- Zone 3: 1.7 g/t Au
- Zone 4: 2.2 g/t Au

The grades were estimated using the inverse distance square (ID2) method with spherical search ellipsoids and a minimum of 2 to a maximum of 12 samples. The radius of the spherical search ellipsoids was as follows:

- Zone 1: 20 m
- Zone 2: 25 m
- Zone 3: 105 m
- Zone 4: 175 m
- Zone 5: 150 m

The specific gravity measurements on 24 samples were used for the resource calculation, yielding an average bulk density of 1.45 g/cm^3.

InnovExplo excluded Zone 5 (Figure 14-11) from the current estimate due to its location (swamps, lake) and the small amount of data.

Table 14-12 displays the results of the tailings portion of the 2022 MRE at a cut-off grade of 0.4 g/t Au. Table 14-13 presents the sensitivity of the tailings portion of the 2022 MRE at different cut-off grades. The reader should be cautioned that the figures provided in this table should not be interpreted as a mineral resource statement. The authors have presented the quantities and grade estimates at different cut-off grades for the sole purpose of demonstrating the sensitivity of the resource model to the selection of a reporting cut-off grade.

The cut-off grade sensitivity analysis (Table 14-13) demonstrates the homogeneity of the gold grade distribution inside the tailings. It is notable that, despite variation of the cut-off grade, there is no real variation in tonnage and grade associated to it. The only variations happen when the cut-off grade tends toward the mean grade of the tailings resource.

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Table 14-12 – Tailings portion of the Duparquet Project 2022 Mineral Resource Estimate

Domain	Cut-off (g/t)	Measured resource			Indicated resource		
		Tonnage (Mt)	Au (g/t)	Ounces	Tonnage (Mt)	Au (g/t)	Ounces
Zones 1 and 2	0.4	19,900	2.03	1,300	-	-	-
Zones 3 and 4		-	-	-	4,105,200	0.93	123,200

Notes to accompany the Mineral Resource Estimate:

1. The independent and qualified persons, as defined by NI 43-101, are Marina Iund, P.Geo., Carl Pelletier, P.Geo., Simon Boudreau, P.Eng., all from Innovexplo and Guy Comeau, P.Eng. from Soutex. The effective date of the estimate is September 12, 2022.
2. These mineral resources are not mineral reserves, as they do not have demonstrated economic viability. There is currently insufficient data to define these Inferred mineral resources as Indicated or Measured mineral resources and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured mineral resource category. The mineral resource estimate follows current CIM Definition Standards.
3. The results are presented in situ and undiluted and have reasonable prospects of economic viability.
4. The estimate encompasses four (4) tailing zones.
5. High-grade capping supported by statistical analysis was done on raw assay data before compositing. High-grade capping was established at 13.0 g/t Au for Zone 1, 3.5 g/t Au for Zone 2, 1.7 g/t Au for Zone 3 and 2.2 g/t Au for Zone 4.
6. The estimate used a block model built in GEOVIA GEMS with a block size of 5m x 5m x 1m. Grades interpolation was obtained by ID2 using hard boundaries.
7. A fixed density of 1.45 g/cm^3 was used in zones and waste.
8. The Measured and Indicated categories were defined based on the drill hole spacing (Measured: zones 1 and 2 = 30m x 30m grid; Indicated: zone 3 = 100m x 100m grid and Zone 4 = 200m x 200m grid).
9. The tailings mineral resource is reported at the in-pit cut-off grade of 0.4 g/t Au. The cut-off grade was calculated using the following parameters: processing cost = CA$11.9; G&A = CA$8.75; refining and selling costs = CA$ 5.00; gold price = US$ 1,650/oz; USD:CAD exchange rate = 1.31; and mill recovery = 93.9%. The cut-off grades should be re-evaluated in light of future prevailing market conditions (metal prices, exchange rates, mining costs etc.).
10. The number of metric tons and ounces was rounded to the nearest hundred, following the recommendations in NI 43-101. Any discrepancies in the totals are due to rounding effects.
11. The authors are not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, or marketing issues, or any other relevant issue not reported in the Technical Report, that could materially affect the Mineral Resource Estimate.

Table 14-13 – Cut-off grade sensitivity for the tailings portion of the Duparquet Project

Domain	Cut-off (g/t)	Measured resource			Indicated resource		
		Tonnage (Mt)	Au (g/t)	Ounces	Tonnage (Mt)	Au (g/t)	Ounces
Zones 1 and 2	0.90	16,000	2.36	1,215			
	0.80	16,800	2.28	1,233			
	0.70	18,000	2.18	1,262			
	0.60	19,000	2.10	1,284			
	0.50	19,400	2.07	1,290			
	0.45	19,600	2.06	1,295			
	0.40	**19,900**	**2.03**	**1,297**			
	0.35	20,000	2.02	1,299			
Zones 3 and 4	0.90				2,424,500	1.01	79,036
	0.80				3,489,000	0.96	108,111
	0.70				4,053,900	0.94	122,102
	0.60				4,104,400	0.93	123,189
	0.50				4,104,800	0.93	123,196
	0.45				4,105,000	0.93	123,200
	0.40				**4,105,200**	**0.93**	**123,203**
	0.35				4,105,400	0.93	123,206

The tailings produced by the former Beattie mine have spread beyond their original location on the Beattie mining concession and a portion now extends onto Crown land (see Figure 14-10). ***There is a risk that the MERN will not allow the issuer to recover the mineral resource contained in the tailings that have spread beyond the Project. However, the authors emphasize that this represents a negligible portion of the Project's total mineral resources. It would have no material impact on the MRE and would not affect the potential viability of the Project.***

14.13.3 Global Duparquet Mineral Resource Estimate

Table 14-14 presents the 2022 Mineral Resource Estimate for the Global Duparquet Project broke down by mining method, at the actual cut off grade.

Table 14-15 presents the sensitivity of the 2022 MRE at different cut-off grades. The reader should be cautioned that the figures provided in this table should not be interpreted as a mineral resource statement. The authors have presented the quantities and grade estimates at different cut-off grades for the sole purpose of demonstrating the sensitivity of the resource model to the selection of a reporting cut-off grade.

Table 14-14 – 2022 Mineral Resource Estimate for the Global Duparquet Project, by mining method

Area (Mining Method)	Cut-off (g/t)	Measured resource			Indicated resource			Inferred resource		
		Tonnage (t)	Au (g/t)	Ounces	Tonnage (t)	Au (g/t)	Ounces	Tonnage (t)	Au (g/t)	Ounces
Open pit	0.4	163,700	1.37	7,200	59,410,600	1.52	2,909,600	28,333,000	1.07	970,400
UG mining	1.5	-	-	-	5,506,900	2.26	399,300	9,038,900	2.29	665,600
Tailings	0.4	19,900	2.03	1,300	4,105,200	0.93	123,200	-	-	-
Total	-	183,600	1.43	8,500	69,022,700	1.55	3,432,100	37,371,900	1.36	1,636,000

Notes to accompany the Mineral Resource Estimate:

12. The independent and qualified persons for the mineral resource estimate, as defined by NI 43-101, are Marina Iund, P.Geo., Carl Pelletier, P.Geo., Simon Boudreau, P.Eng., all from Innovexplo and Guy Comeau, P.Eng. from Soutex. The effective date of the estimate is September 12, 2022.
13. These mineral resources are not mineral reserves, as they do not have demonstrated economic viability. There is currently insufficient data to define these Inferred mineral resources as Indicated or Measured mineral resources and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured mineral resource category. The mineral resource estimate follows current CIM Definition Standards.
14. The results are presented in situ and undiluted and have reasonable prospects of economic viability.
15. In-pit and Underground estimates encompass sixty (60) mineralized domains and one dilution envelop using the grade of the adjacent material when assayed or a value of zero when not assayed; The tailings estimate encompass four (4) zones.
16. In-pit and Underground: High-grade capping of 25 g/t Au; Tailings: High-grade capping of 13.0 g/t Au for Zone 1, 3.5 g/t Au for Zone 2, 1.7 g/t Au for Zone 3 and 2.2 g/t Au for Zone 4. High-grade capping supported by statistical analysis was done on raw assay data before compositing.
17. In-pit and Underground: The estimate used a sub-block model in GEOVIA SURPAC 2021 with a unit block size of 5m x 5m x 5m and a minimum block size of 1.25m x 1.25m x1.25m. Grade interpolation was obtained by ID2 using hard boundaries. Tailings: The estimate used a block model in GEOVIA GEMS with a block size of 5m x 5m x 1m. Grade interpolation was obtained by ID2 using hard boundaries.
18. In-pit and Underground: A density value of 2.73 g/cm^3 was used for the mineralized domains and the envelope. A density value of 2.00 g/cm^3 was used for the overburden. A density value of 1.00 g/cm^3 was used for the excavation solids (drifts and stopes) assumed to be filled with water. Tailings: A fixed density of 1.45 g/cm^3 was used in zones and waste.
19. In-pit and Underground: The mineral resource estimate is classified as Measured, Indicated and Inferred. The measured category is defined by blocks having a volume of at least 25% within an envelope built at a distance of 10 m around existing channel samples. The Indicated category is defined by blocks meeting at least one (1) of the following conditions: Blocks falling within a 15-m buffer surrounding existing stopes and/or blocks for which the average distance to composites is less than 45 m. A clipping polygon was generated to constrain Indicated resources for each of the sixty (60) mineralized domains. Only the blocks for which reasonable geological and grade continuity have been demonstrated were selected. All remaining interpolated blocks were classified as Inferred resources. Blocks interpolated in the envelope were all classified as Inferred resources. Tailings: The Measured and Indicated categories were defined based on the drill hole spacing (Measured: Zones 1 and 2 = 30m x 30m grid; Indicated: Zone 3 = 100m x 100m grid and Zone 4 = 200m x 200m grid).
20. In-pit and Underground: The mineral resource estimate is locally pit-constrained with a bedrock slope angle of 50° and an overburden slope angle of 30°. The out-pit mineral resource met the reasonable prospect for eventual economic extraction by having constraining volumes applied to any blocks (potential underground extraction scenario) using DSO. It is reported at a rounded cut-off grade of 0.4 g/t Au (in-pit and tailings) and 1.5 g/t Au (UG). The cut-off grades were calculated using the following parameters: mining cost = CA$70.00 (UG); processing cost = CA$11.9 to 17.0; G&A = CA$8.75; refining and selling costs = CA$ 5.00; gold price = US$ 1,650/oz; USD:CAD exchange rate = 1.31; and mill recovery = 93.9%. The cut-off grades should be re-evaluated in light of future prevailing market conditions (metal prices, exchange rates, mining costs etc.).
21. The number of metric tons and ounces was rounded to the nearest hundred, following the recommendations in NI 43-101. Any discrepancies in the totals are due to rounding effects.
22. The authors are not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, or marketing issues, or any other relevant issue not reported in the Technical Report, that could materially affect the Mineral Resource Estimate.

Table 14-15 – Cut-off grade sensitivity for the Duparquet Project

Area (mining method)	Cut-off (g/t)	Stripping Ratio	Measured resource			Indicated resource			Inferred resource		
			Tonnage (Mt)	Au (g/t)	Ounces	Tonnage (Mt)	Au (g/t)	Ounces	Tonnage (Mt)	Au (g/t)	Ounces
Open pit	0.7	6.35	137,321	1.53	6,755	23,142,210	2.05	1,525,279	2,592,695	1.62	135,038
	0.65	6.32	141,757	1.5	6,836	25,666,698	1.98	1,633,902	3,334,098	1.48	158,647
	0.6	6.73	149,158	1.46	7,001	32,690,577	1.86	1,954,908	5,716,620	1.34	246,283
	0.55	6.53	154,634	1.42	7,060	36,556,977	1.77	2,080,340	7,727,020	1.23	305,568
	0.5	6.53	156,938	1.41	7,122	41,152,335	1.70	2,253,068	11,007,061	1.13	400,881
	0.45	7.69	161,081	1.39	7,187	53,548,726	1.58	2,722,586	22,032,449	1.16	824,601
	0.4	7.56	163,709	1.37	7,222	59,410,612	1.52	2,909,551	28,332,980	1.07	970,424
	0.35	7.44	165,800	1.36	7,248	66,307,600	1.46	3,117,172	37,354,222	0.96	1,147,282
UG mining	1.9	-	-	-	-	5,891,904	2.67	505,871	7,168,869	2.91	669,750
	1.7	-	-	-	-	5,224,787	2.47	414,153	7,378,504	2.51	595,956
	1.5	-	-	-	-	5,506,861	2.26	399,356	9,038,871	2.29	665,629
	1.3	-	-	-	-	5,302,381	2.10	357,603	11,459,118	2.05	756,440
Tailings	0.6	-	19,000	2.10	1,284	4,104,400	0.93	123,189	-	-	-
	0.5	-	19,400	2.07	1,290	4,104,800	0.93	123,196	-	-	-
	0.45	-	19,600	2.06	1,295	4,105,000	0.93	123,200	-	-	-
	0.4	-	19,900	2.03	1,297	4,105,200	0.93	123,203			
	0.35	-	20,000	2.02	1,299	4,105,400	0.93	123,206			

15. MINERAL RESERVE ESTIMATES

Not applicable at the current stage of the Project.

16. MINING METHODS

Not applicable at the current stage of the Project.

17. RECOVERY METHODS

Not applicable at the current stage of the Project.

18. PROJECT INFRASTRUCTURE

Not applicable at the current stage of the Project.

19. MARKET STUDIES AND CONTRACTS

Not applicable at the current stage of the Project.

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

Not applicable at the current stage of the Project.

21. CAPITAL AND OPERATING COSTS

Not applicable at the current stage of the Project.

22. ECONOMIC ANALYSIS

Not applicable at the current stage of the Project.

23. ADJACENT PROPERTIES

The Project is located within an active exploration area of the southern Abitibi greenstone belt. It covers a significant portion of the Duparquet sector in the Rouyn-Noranda mining district.

The issuer holds 100% interest on two nearby projects:

- The Pitt Gold project owned by Mines d'Or Duquesne Gold Mines. In 2017, a NI 43-101 MRE reported an inferred resources of 1,076,000 t at 7.42 g/t Au for a total of 257,000 oz (Lewis & San Martin, 2017).
- The Duquesne project owned by Mines d'Or Duquesne Gold Mines and Clifton Star Ressources Inc. In 2016, a NI 43-101 MRE reported an indicated resources of 1,859,200 t at 3.33 g/t Au for a total of 199,161 oz and an inferred resources of 1,563,100 t at 5.58 g/t Au for a total of 280,643 oz (Rioux, 2016).

The author has not verified any mineral resource estimates or published geological information pertaining to the adjacent properties.

Figure 23-1 shows the owners of adjacent properties. The author has not verified any of the published mineral resource estimates or geological information pertaining to the adjacent properties. Information regarding mineralization on adjacent properties is not necessarily indicative of mineralization on the Project.

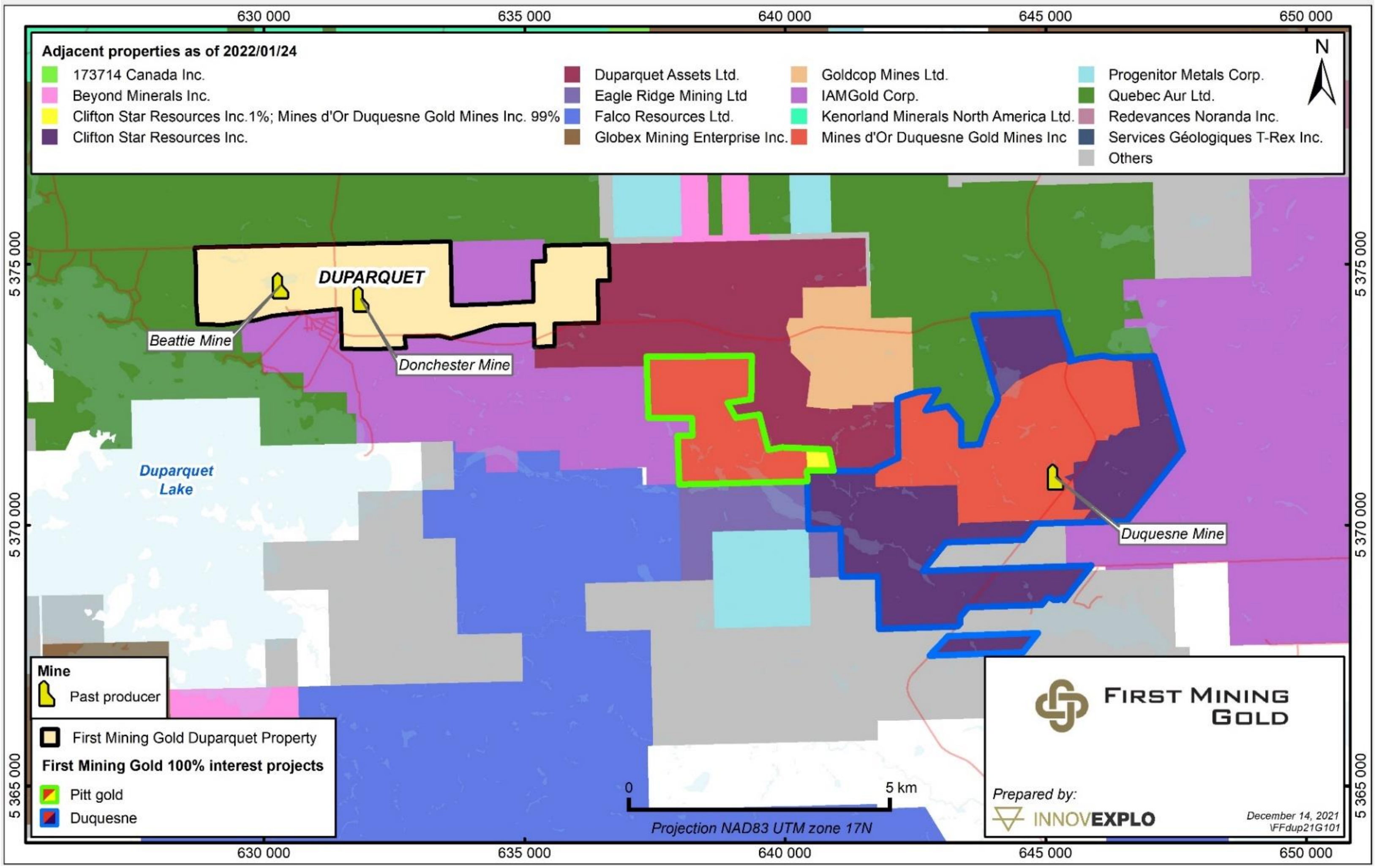

Figure 23-1 – Adjacent properties

24. OTHER RELEVANT DATA AND INFORMATION

Silver has not been included in any of the mineral resource estimates for the Project because few silver assays are available.

The sections below present the silver data known to the authors.

24.1 2009 Sampling of the Beattie Mine Tailings (silver)

Eight (8) samples from the tailings were assayed for silver in 2009. The average silver grade of all results was 2.4 g/t Ag (Fillion, 2009).

24.2 2013 Bulk Sample (silver)

In 2013, Clifton Star sent a 12 t composite bulk sample to SGS. The results yielded a head analysis of 1.84 g/t Au and 2.2 g/t Ag, and the Au/Ag ratio was 0.84.

The 2013 bulk sample comprises material from six (6) mineralized zones. A total of 238 pulp samples were assayed for silver. The silver grades are plotted against Au/Ag ratios in Figure 24-1.

24.3 Historical Production of Silver from the Beattie Mine

From the start of production in 1933 until the end of 1940, the mill at the Beattie gold mine processed 3,921,281 short tons of mineralized material and recovered 471,085 ounces of gold and 73,214 ounces of silver, for an average of 0.120 oz/ton Au and 0.019 oz/ton Ag (Dresser and Denis, 1949). This tonnage came from the North and A zones. The Au/Ag ratio was 6.32.

According to the Department of Energy and Natural Resources of Quebec (Lavergne, 1985), the Beattie mill treated 9,645,000 t with an average grade of 4.01 g/t Au and 0.99 g/t Ag. The mineralized material came from the Beattie mine and Donchester mine between 1933 and 1956. The Au/Ag ratio decreased to 4.05 during this time because the mineralized material from the North Zone of the Donchester mine was added to the Beattie mill. This is also evident in Figure 24-1, where the North Zone in the Donchester mine (green diamonds; DDH D09-18) shows a higher silver content than that of the North Zone in the Beattie mine (beige diamonds; DDH BD12-15).

Silver recovery was not optimal during these years. In reality, the historical Au/Ag ratio was probably lower than Dresser and Denis (1949) and Lavergne (1985) reported.

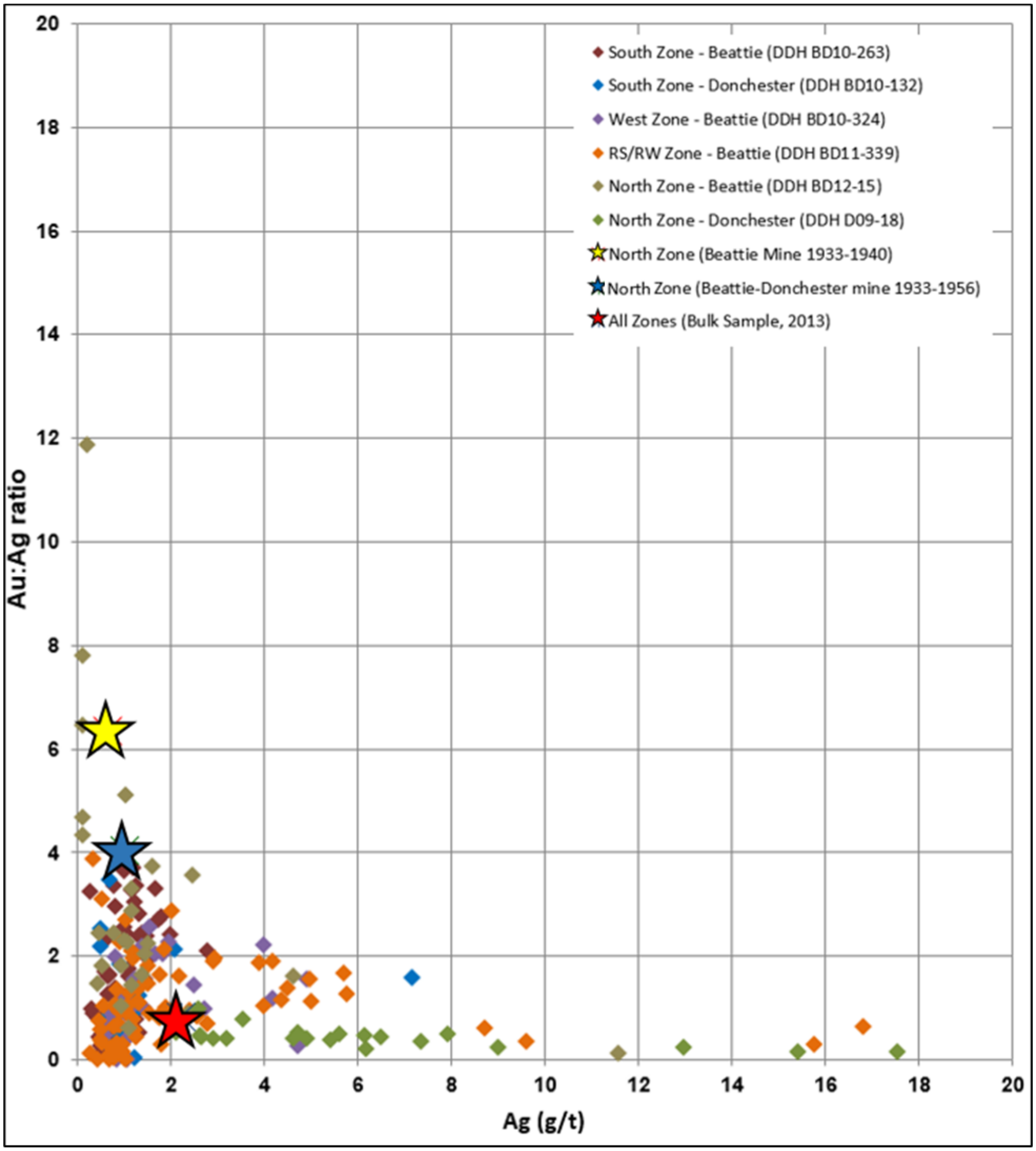

Figure 24-1 – Plot of silver grades (g/t Ag) vs. Au/Ag ratios of assay results from the 2013 bulk sample and from historical production of the Beattie mine

25. INTERPRETATION AND CONCLUSIONS

The objective of InnovExplo's mandate was to provide an updated mineral resource estimate for the Project (the "2022 MRE"). The mandate covers the historical Beattie gold-producing mining concession and its associated tailings disposal area, the historical Donchester concession, and the Dumico and Central Duparquet claim blocks. This Technical Report and the 2022 MRE herein meet this objective.

The authors conclude the following:

- The database supporting the 2022 MRE is complete, valid and up to date.
- Geological and gold-grade continuity has been demonstrated for all 72 mineralized zones.
- The key parameters of the 2022 MRE (density, capping, compositing, interpolation, search ellipsoid, etc.) are supported by data and statistical and/or geostatistical analysis.
- The 2022 MRE includes measured, indicated and inferred resources for a combination of two mining scenarios: open pit and selective underground. The 2022 MRE complies with CIM Definition Standards and CIM Guidelines.
- Two cut-off grades of 0.40 and 1.50 g/t Au were used, corresponding to potential open pit and selective underground mining scenarios.
- Cut-off grades were calculated at a gold price of US$1,650 per troy ounce and an exchange rate of 1.31 USD/CAD, using reasonable mining, processing and G&A costs.
- In a combined pit and selective underground mining scenario, the Project contains an estimated M+I Resource of 65,081,200 t at 1.58 g/t Au for 3,316,100 oz of gold and an Inferred Resource of 37,371,900 t at 1.36 g/t Au for 1,636,000 oz of gold. The Project also contains the Beattie mine tailings with an estimated M+I Resource of 4,125,100 t at 0.94 g/t Au for 124,500 oz of gold.
- The results of the 2022 MRE represent a 10.5% increase in the M+I Resource and a 13.4% increase in the Inferred Resource compared to the previous 2014 MRE of Poirier et al., 2014 (Table 25-1). The increase in the M+I Resource is due to a deeper optimized shell and the updated economic parameters. The same reasons combined with the addition of 55 drill holes explain the increase in Inferred resources.
- Based on metallurgical tests, the Duparquet project appears amenable to existing gold recovery processes. A combination of flotation, pressure oxidation and cyanide leach processes has shown a gold recovery ranging from 94.7% to 96.5%.
- Additional diamond drilling on multiple zones would likely upgrade some of the Inferred Resource to the Indicated category and/or add to the Inferred Resource since most of the mineralized zones have not been fully explored at depth or close to surface infrastructures.

At this stage, it is reasonable to believe that a hybrid operation consisting of an early open pit followed by later underground mining activities is amenable to the expectation of "reasonable prospects of eventual economic extraction" as stated in the CIM Guidelines. The potential to add new resources in the open pit through exploration is best focused in the east direction because the favorable geology hosting the Project mineralization is constrained to the west, and the pit itself is constrained to the south by the Project limit and the town of Duparquet. Drilling to tighten the drill spacing in the inferred resources will permit a transfer from inferred to indicated by adding confidence in the estimate. There is potential to add material at depth below the existing mineralized model that could be accessed from the underground infrastructures. The reader is cautioned that this exploration target is not a mineral resource estimate and is conceptual in nature. There has been insufficient exploration to define it as a mineral resource and it is uncertain if further exploration will delineate the exploration target as a mineral resource.

The authors consider the 2022 MRE reliable, thorough, and based on quality data, reasonable hypotheses, and parameters compliant with NI 43-101 requirements, CIM Definition Standards and CIM Guidelines.

The authors consider the 2022 MRE reliable, thorough, and based on quality data, reasonable hypotheses, and parameters compliant with NI 43-101 requirements, CIM Definition Standards and CIM Guidelines.

25.1 Mineral Resource Estimates

Table 25-1 compares the Project Mineral Resources statement of September 12, 2022 to the statement from 2014.

Table 25-1– Measured, Indicated and Inferred Mineral Resources between 2022 and 2014

Year	Category	Tonnage	Grade (g/t)	Gold ounces
2022	M + I	69,206,300	1.55	3,440,600
	INF	37,371,900	1.36	1,636,000
2014	M + I	60,881,000	1.59	3,113,171
	INF	29,684,700	1.51	1,442,689

Several factors may affect the mineral resource estimate, including metal price, exchange rate (CAD:USD), , data on which engineering assumptions are made that prove faulty, or environmental and socioeconomics considerations.

25.2 Risks and Opportunities

Table 25-2 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, government regulations, etc.).

Significant opportunities that could improve the economics, timing and permitting are identified in Table 25-3. Further information and study are required before these opportunities can be included in the project economics.

Table 25-2 – Risks for the Project

Risk	Potential impact	Possible risk mitigation
Difficulty in attracting experienced professionals	The ability to attract and retain competent, experienced professionals is a key success factor	The early search for professionals will help identify and attract critical people.
Downsizing of the optimized resource shell due its proximity with the municipality of Duparquet and private lands	A negative transfer of low-cost open pit resources toward higher-cost underground resources	Relocation of houses away from the buffer zone of the pit
Recovery of existing tailings on government-owned land	Loss of tailing materials	Proactive discussion with Quebec regulators centred on ground reclamation
Quebec Ministry of Transportation will not allow a by-pass of highway 393	Loss of resources inside a potential permanent pillar within the pit	Proactive discussion with Quebec regulators

Table 25-3 – Opportunities for the Project

Opportunities	Explanation	Potential benefit
Experienced workforce	An experienced workforce already presents in the Abitibi region	Creation of a team-building environment
Exploration at depth	Zones are still open at depth	Increase in underground mineral resources
Pursue other metallurgical recovery processes	BIOX and ALBION processes were tested and evaluated in the past, but recent advancements in these technologies make them more attractive now	Replacement of acid leaching with a bacteria solution or higher oxidation following a finer grinding

26. RECOMMENDATIONS

26.1 Geology

The authors recommend further exploration drilling on the Project to potentially increase resources and the confidence level of the geological model. More specifically, the NE-SW striking secondary zones should be drilled to test their lateral and depth extensions. Additional detailed surface mapping and channel sampling would enhance the structural model. A compilation of historical diamond drill holes on the Central Duparquet Property is recommended, particularly those drilled during the 1980s to cover the current mineralized zones in that area. Further definition drilling is recommended along strike and at depth to upgrade the Inferred resources to the Indicated category and address the underground potential for all zones.

26.2 Mining

Additional geotechnical and hydrogeological studies should be undertaken to better define the pit wall slopes presented in this report. This would involve confirming the structural data over the proposed footprint of the open pit. Ideally, this would involve a geotechnical drilling program with a minimum of one (1) hole oriented perpendicular to each of the four pit walls (north, south, east and west).

26.3 Metallurgy

The QP recommends the following tests:

- Additional variability hardness tests throughout the Project. Several samples from each zone should undergo SMC, rod mill and ball mill Bond tests.
- A variability locked-cycle testing program for the existing tailings area involving a mix of tailings and mineralized material.
- The metallurgical results using the POX process have been positive. However, the QP believes that the BIOX and Albion processes should be re-investigated.

26.4 Costs Estimate for Recommended Work

The authors have prepared a cost estimate for the recommended two-phase work program to serve as a guideline. The budget for the proposed program is presented in Table 26-1. Expenditures for Phase 1 are estimated at C$4.65 million (incl. 15% for contingencies). Expenditures for Phase 2 are estimated at C$1.25 million (incl. 15% for contingencies). Phase 2 is contingent upon the success of Phase 1. The grand total is C$5,9 million (incl. 15% for contingencies).

Table 26-1 – Estimated Costs for the Recommended Work Program

Phase 1	Work program	Budget cost
a	Drilling and Compilation of historical drill holes	$2.5 M
b	Resource Update	$150 k
c	Geotechnical Drilling	$1.0 M
d	Metallurgical testwork	$1.0 M
	Phase 1 subtotal	$4.65 M
Phase 2	**Work program**	**Budget cost**
a	Environmental Assessment	$800 K
b	Hydrogeological Study	$250 K
	Phase 2 subtotal	$1.05 M
	TOTAL (Phase 1 and Phase 2)	$5.7 M

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