



NI 43-101 Technical Report Preliminary Economic Assessment Cadillac Project, Quebec, Canada

Effective Date: August 31, 2026
Issued Date: October 30, 2026



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PLR Resources Inc.

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Certificate of Qualified Person

Alexandre Burelle, P. Eng.

This certificate applies to the NI 43-101 Technical Report entitled Preliminary Economic Assessment, Cadillac Project, Québec, Canada (the “Technical Report”), prepared for Cartier Resources Inc., with an effective date of August 31st, 2026, and an issued date of October 30th, 2026.

I, Alexandre Burelle, P. Eng., as a co-author of the Technical Report, do hereby certify that:

1. I am a Mining Engineer and consultant for Evomine Consulting Inc. with an address of 419 rue des Hirondelles, Beloeil, Quebec, Canada, J3G 6G8.
2. I graduated from McGill University, Montreal, Quebec, Canada, with a B.Eng. in Mining Engineering in 2012 and from Imperial College London, London, United Kingdom, with a M.Sc. in Metals and Energy Finance in 2013.
3. I am a professional engineer in good standing with the Ordre de ingénieurs du Québec (OIQ) in Canada (no. 5019855).
4. My relevant experience for the purpose of the Technical Report is over twelve years of experience in mining operations, technical study delivery, due diligence, mine financing, business development, and strategic development.
5. I have read the definition of “qualified person” set out in the NI 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections of the Technical Report: 1.11, 1.13, 1.16, 1.17, 15, 16, 18, 21 (except 21.3.6, 21.4.1, 21.5.1, 21.7.2 and 21.7.5.2), 22, 25.7, 25.9, 25.10, 27.
8. I have visited the Property that is the subject of the Technical Report, on September 18th, 2025, as part of this current mandate.
9. I have had no prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared following NI 43-101 rules and regulations.
11. As at 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 portions of the Technical Report for which I am responsible not misleading.

Signed and sealed on this 30th day of October 2026.

(Original signed and sealed)





Certificate of Qualified Person

Stephen Coates, P. Eng.

This certificate applies to the NI 43-101 Technical Report entitled Preliminary Economic Assessment, Cadillac Project, Québec, Canada (the “Technical Report”), prepared for Cartier Resources Inc., with an effective date of August 31st, 2026, and an issued date October 30th, 2026.

I, Stephen Coates, P. Eng., as a co-author of the Technical Report, do hereby certify that:

1. I am a Mining Engineer and consultant for Evomine Consulting Inc. with an address of 419 rue des Hirondelles, Beloeil, Quebec, Canada, J3G 6G8.
2. I graduated from McGill University, Montreal, Quebec, Canada, with B.Eng. in Mining Engineering in 2013.
3. I am a professional engineer in good standing with the Ordre de ingénieurs du Québec (OIQ) in Canada (no. 5047905).
4. My relevant experience for the purpose of the Technical Report is over ten years of experience in mining operations, technical study delivery, due diligence, mine financing, business development, and strategic development.
5. I have read the definition of “qualified person” set out in the NI 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections of the Technical Report: 1.1, 1.2, 1.3, 1.14, 1.18, 1.19, 2, 3, 14.14, 19, 25.1, 25.11, 25.12, 26, 27.
8. I have visited the Property that is the subject of the Technical Report, on September 18, 2025, as part of this current mandate.
9. My prior involvement with the property includes internal trade-off studies on the project and ongoing mining engineering support.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared following NI 43-101 rules and regulations.
11. As at 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 portions of the Technical Report for which I am responsible not misleading.

Signed and sealed on this 30th day of October 2026.

(Original signed and sealed)





Certificate of Qualified Person

Maude Lévesque Michaud, P.Eng., M.Sc.A.

This certificate applies to the NI 43-101 Technical Report entitled Preliminary Economic Assessment, Cadillac Project, Québec, Canada (the “Technical Report”), prepared for Cartier Resources Inc., with an effective date of August 31st, 2026, and an issued date October 30th, 2026.

I, Maude Lévesque Michaud, P. Eng., M. Sc. A., as a co-author of the Technical Report, do hereby certify that:

1. I am a Geological Engineer and consultant for Geodoz conseil Inc. with an address of 51 rue Quidoz, Sainte-Thérèse, Québec, Canada, J7E 4L3.
2. I graduated from Laval University, Quebec City, Quebec, Canada, with B.Eng. in Geological Engineering in 2010 and from University of Quebec in Abitibi-Témiscamingue, Rouyn-Noranda, Quebec, Canada, with a M.Sc.A. in Mineral Engineering in 2016.
3. I am a professional engineer in good standing with the Ordre de ingénieurs du Québec (OIQ) in Canada (no. 5015957).
4. My relevant experience for the purpose of the Technical Report is over fifteen years of experience in mining exploration, environmental studies, permitting, geochemical characterization of mining materials and technical study delivery.
5. I have read the definition of “qualified person” set out in the NI 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections of the Technical Report: 1.15, 1.16, 20, 21.5.1, 25.8, 27.
8. I have not visited the Property that is the subject of the Technical Report.
9. I have had no prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared following NI 43-101 rules and regulations.
11. As at 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 portions of the Technical Report for which I am responsible not misleading.

Signed and sealed on this 30th day of October 2026.

(Original signed and sealed)





Certificate of Qualified Person

Pierre-Luc Richard, P.Geo.

This certificate applies to the NI 43-101 Technical Report entitled Preliminary Economic Assessment, Cadillac Project, Québec, Canada (the “Technical Report”), prepared for Cartier Resources Inc., with an effective date of August 31st, 2026, and an issued date October 30th, 2026.

I, Pierre-Luc Richard, P. Geo., as a co-author of the Technical Report, do hereby certify that:

1. I am a professional geologist at the consulting firm PLR Resources Inc., located at 2000 McGill College Avenue, Suite 600, Montreal, Quebec, Canada H3A 3H3.
2. I am a graduate of Université du Québec à Montréal in Resource Geology (2004). I also obtained an M.Sc. from Université du Québec à Chicoutimi in Earth Sciences in 2012.
3. I am a member in good standing of the Ordre des Géologues du Québec (OGQ No. 1119), the Professional Geoscientists of Ontario (APO No. 1714), and the Northwest Territories Association of Professional Engineers and Geoscientists (NAPEG No. L2465).
4. I have worked in the mining industry for more than 20 years. My exploration and mining expertise has been acquired with numerous companies throughout my career. I managed and QP'd numerous technical reports, mineral resource estimates, and audits as a consultant with different firms and for PLR Resources since 2022.
5. I have read the definition of “qualified person” set out in the NI 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections of the Technical Report: 1.4, 1.5, 1.6, 1.7, 1.9, 1.10, 4, 5, 6, 7, 8, 9, 10, 11, 12, 14 (except 14.14), 23, 24, 25.2, 25.3, 25.4, 25.5, 27.
8. I have visited the Property that is the subject of the Technical Report, on May 26, 2025, as part of this current mandate.
9. I have had no prior involvement with the property that is the subject of the Technical Report, except to provide limited technical support regarding drilling planning.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared following NI 43-101 rules and regulations.
11. As at 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 portions of the Technical Report for which I am responsible not misleading.

Signed and sealed on this 30th day of October 2026.

(Original signed and sealed)





Certificate of Qualified Person

Pierre Roy, P. Eng., M. Sc.

This certificate applies to the NI 43-101 Technical Report entitled Preliminary Economic Assessment, Cadillac Project, Québec, Canada (the “Technical Report”), prepared for Cartier Resources Inc., with an effective date of August 31st, 2026, and an issued date October 30th, 2026.

I, Pierre Roy, P.Eng., M.Sc., as a co-author of the Technical Report, do hereby certify that:

1. I am currently a Senior Metallurgist and Specialist–Mineral Processing at Soutex inc. located at 1990 rue Cyrille-Duquet, Local 204, Quebec, Province of Quebec, Canada G1N 4K8..
2. I graduated from the Université Laval (Quebec, Quebec, Canada) with a B.Sc. in Mining Engineering in 1986, and a M.Sc. in Mining in 1989.
3. I am a registered member in good standing with Ordre des ingénieurs du Québec, licence no. 45201.
4. I have practiced my profession continuously in the mining industry since my graduation from university. I have been involved in mining operations, engineering and financial evaluations for 38 years. During this time, I have been involved in mineral processing and environmental coordination at Kiena mine for six (6) years and Troilus mine for nine (9) years. I have also worked as consultant for the mineral processing industry for two (2) years at CRM in Québec and with Soutex inc. in Québec for twenty-one (21) years. As consultant I have been involved in many projects with Iron, base metals and gold mining sectors.
5. I have read the definition of “qualified person” set out in the NI 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections of the Technical Report: 1.8, 1.12, 1.16, 13, 17, 21.3.6, 21.4.1, 21.7.2, 21.7.5.2, 25.6, 27.
8. I have not visited the Property that is the subject of the Technical Report.
9. I have had no prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared following NI 43-101 rules and regulations.
11. As at 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 portions of the Technical Report for which I am responsible not misleading.

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1 SUMMARY

1.1 Introduction

Cartier Resources Inc. (“Cartier”, the “Company” or the “Issuer”) mandated Evomine Consulting Inc. (“Evomine”) to lead a group of consulting firms, including Geodoz Conseil Inc. (“Geodoz”), PLR Resources (“PLR”) and Soutex Inc. (“Soutex”) to prepare an updated Preliminary Economic Assessment (“PEA”) for the Cadillac Project (the “Project”).

The purpose of this Technical Report is to present the results of the updated PEA for the Project. The updated PEA evaluates the potential technical and economic viability of developing the Project based on the current mineral resource estimate and a conceptual mine development plan.

The assessment considers the principal technical and economic aspects of the Project, including mining methods, mineral processing and metallurgical recovery, project infrastructure, environmental and permitting considerations, capital and operating cost estimates, and an economic analysis based on the assumptions and parameters described in this Report.

This Technical Report has been prepared in accordance with National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* (“NI 43-101”) and Form 43-101F1 – *Technical Report* and is intended to provide a summary of the material scientific and technical information supporting the updated PEA.

The Project is located in Abitibi-Témiscamingue, Québec, Canada, approximately 35 km east-southeast of the city of Val-d’Or. Cartier is a Canadian publicly traded company listed on the TSX Venture Exchange (“TSXV”) under the trading symbol ECR, with its head office located in Val-d’Or, Québec.

1.2 Terms of Reference

This Report supports the disclosure in Cartier’s news release dated September 17, 2026, titled “Cartier announces updated PEA for the Cadillac Project: After-tax NPV5% of C\$1.0 billion and after-tax IRR of 26.6% at US\$3,600/oz gold price”. The effective date of the Report is August 31, 2026.

All measurement units presented in this Technical Report, unless noted otherwise, in this Report are metric. Currency is expressed in Canadian dollars (“CAD” or “\$”) unless otherwise noted. Costs are based on third quarter (“Q3”) 2026 dollars.



As of the effective date of this Report, the authors of this Report (the “Report Authors”) are not aware of any known litigation potentially affecting the Project. The Qualified Persons (“QPs”) did not verify the legality or terms of any underlying agreement(s) that may exist concerning the Project ownership, permits, off-take agreements, license agreements, royalties or other agreement(s) between Cartier and any third parties.

The opinions contained herein are based on information collected during investigations by the QPs, which in turn reflects various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results can be significantly more or less favourable.

1.3 Contributors

The following individuals, by virtue of their education, experience and professional association, are considered QPs as defined in the NI 43-101 and are members in good standing of appropriate professional institutions:

- Alexandre Burelle, P.Eng. (Evomine Consulting Inc.)
- Stephen Coates, P.Eng. (Evomine Consulting Inc.)
- Maude Lévesque Michaud, P.Eng, M.Sc.A. (Geodoz Conseil Inc.)
- Pierre-Luc Richard, P.Geo. (PLR Resources Inc.)
- Pierre Roy, P.Eng. (Soutex Inc.)

1.4 Property Description, Location and Ownership

The Cadillac Project is located approximately 35 km east-southeast of the town of Val-d’Or, in the Province of Québec, Canada and approximately 5 km southeast of the municipality of Louvicourt (Figure 4-1). The Project is accessible year-round via paved roads and secondary gravel roads.

The coordinates for the approximate centre of the Project are latitude 48°00' N and longitude 77°06' W (343350E and 5318430N: NAD 83 / UTM Zone 18N) on NTS map sheets 32C03 and 31N14.

Cartier controls a group of 281 mining titles covering a total of 11,596 ha. This information is current as of the effective date of this Report. Figure 4-2 shows the mining titles of the Cadillac Project. Cartier’s asset integrates the historic Chimo Mine and East Cadillac projects.

The mining titles are recorded under Cartier Resources Inc. and are in good standing as of the effective date of this Report.



Numerous mining titles comprising the Project are subject to certain agreements and royalties, varying from 1 to 2% of Net Smelter Return (“NSR”) and 3% Gross Metal Return (“GMR”).

1.5 Geology

The Cadillac Project is located in the Southern Volcanic Zone of the Abitibi Subprovince. This zone is crossed in the north by the possible extension of the Larder Lake-Cadillac deformation zone.

The project is crossed by the volcano-sedimentary rocks of the Trivio Complex. The Trivio Complex is composed of a structural complex containing a set of lenticular bands of sedimentary rocks and mafic volcanic rocks, in shear contact with each other, without it being possible to link them stratigraphically.

Sedimentary rocks form a rhythmic sequence of proximal turbidites, composed of fine quartz-feldspar sandstones and siltstones, interbedded with levels of magnetite-bearing iron formations, coarse feldspar sandstones, and a few levels of petromict conglomerates.

Volcanic rocks are grouped within two main belts (north and south volcanic belts) and consist mainly of massive, pillow, and, more rarely, brecciated basaltic and andesitic flows. Volcaniclastic and exhalative rock horizons interdigitate with sedimentary rocks, indicating that volcanism was active during sedimentation.

Two gold deposits (the historic Chimo Mine and the West Nordeau occurrence) and several occurrences are known on the Project. The Chimo mine produced 379,012 ounces of gold through three producers between 1964 and 1997. The West Nordeau deposit lies 1.5 km east of the Chimo Mine.

Gold mineralization in the area is typically concentrated along the LLCFZ and related secondary structures. This is evident at the closed Chimo Mine and the West Nordeau deposit, where gold occurs with quartz and arsenopyrite in longitudinal high-strain (“shear”) zones within the mafic volcanic rocks and in bands of semi-massive arsenopyrite and pyrrhotite associated with banded magnetite iron formation units.

Gold mineralization on the Project occurs epigenetically in silicified lodes with disseminated sulphides, spatially related to banded iron formation and altered shear zones with temporally related quartz ± carbonate veins. When related to shear zones, Au mineralization typically occurs in volcanic units with disseminated arsenopyrite, pyrite and chalcopyrite. Graphite horizons are also common with this type of mineralization. In addition, several new prospective gold zones have been discovered in recent years, namely the South Nordeau, North Contact, Far West Simon and Lac

Rapides zones, all of which are roughly parallel to, and closely associated with, the LLCFZ.

1.6 Drilling

Cartier initiated its first drilling program on November 1, 2016. Drilling was divided into five phases on the Cadillac property. The objective was to expand the known gold zones and enhance the discovery potential for new gold zones. A total of 286 Drill holes and 85,980 m were drilled by Cartier.

1.7 Data Verification

For the purpose of this MRE, PLR performed a basic verification on the entire Project database. All data was provided by Cartier in UTM coordinates. The database close-out date for the resource estimate is February 17, 2025; data from 4,477 diamond drillholes (DDH) (535,360 m) and 167,978 assays is part of the database.

The QP visited the Project in May 2025, during the course of this mandate. The site visit included a visual inspection of historical core drilled by past owners and recent core drilled by the issuer, as well as a field tour and discussions of geological interpretations with the issuer's geologists and management. The site visit also included a review of sampling and assaying procedures, the quality assurance / quality control ("QA/QC") program, downhole survey methodologies, and the descriptions (logging) of lithologies, alteration and structures. Selected drill collars were also validated in the field using a handheld GPS.

The QP also reviewed sections of mineralized core while visiting the Project. All mineralized core boxes were labelled and properly stored. Sample tags were present in the boxes, and it was possible to validate sample numbers and confirm the presence of mineralization in witness half-core samples from the mineralized zones. The sample preparation room visited was clean and no issues were noted. Analytical, QA/QC or security procedures are conducted in accordance with the industry standards.

The QP is of the opinion that the drilling protocols in place are adequate. The database for the Cartier Project is of good overall quality. No material issues were noted during the validation process. In the QP's opinion, the Cartier database is appropriate to be used for the estimation of Mineral Resources.

1.8 Mineral Processing and Metallurgical Testing

One metallurgical test program has been completed on samples from the Cadillac Project. The



program was carried out by ALS Metallurgy under job number KM7798, between December 2025 and September 2026, on nine composites representing the three deposits considered in this study: Chimo, East Chimo and West Nordeau.

- 2025–2026 testwork on nine (9) composites:
 - Gold grade by metallic screen assay, with duplicate assays on all leach residues;
 - Bulk mineralogy and gold association on each composite;
 - Whole rock analysis and sulphur determination;
 - Grindability by SMC test and Bond ball mill work index;
 - Whole-rock cyanide leaching at three grind sizes per composite (K80 of 46 to 128 μm);
 - Whole-rock cyanide leaching under modified conditions: pre-aeration, lead nitrate addition and elevated pH;
 - Gravity separation by Knelson concentrator with Mozley table upgrading;
 - Cyanide leaching on gravity tails;
 - Cyanide destruction on cyanidation tails.

Forty-four (44) tests were completed in total: six (6) gravity separation tests, thirty-two (32) whole-rock cyanide leach tests and six (6) cyanide leach tests on gravity tails and cyanide detoxification on the tails of those tests. This program was designed specifically around the mineral content considered in the present study, and its results form the basis of the recovery, grind size and reagent consumption assumptions adopted herein.

The results support the following conclusions:

- Mineralogy is consistent between the three deposits, with moderate sulphide contents and no deleterious minerals identified. Pyrrhotite is more abundant in Chimo and East Chimo, and arsenopyrite in West Nordeau. The variability composites show limited intra-deposit variability, which supports the representativeness of the master composites;
- Comminution characteristics are comparable across the three deposits, with Bond ball mill work indices of 13.1 kWh/t to 14.3 kWh/t and SCSE values of 11.3 kWh/t to 13.1 kWh/t. These values indicate moderate to high competency relative to the JKTech database, but all three deposits are expected to respond comparably in a conventional crushing and grinding circuit;
- Gold is amenable to whole-rock cyanidation. Under standard conditions, recovery ranges



from 90.1% to 98.0% at a K80 of 48 μm to 56 μm , and decreases to 77.6% to 87.7% at a K80 above 120 μm , confirming a marked sensitivity to grind size;

- Lead nitrate addition combined with pre-aeration accelerates gold dissolution and, at the finest grind size, reduces average cyanide consumption from 0.92 kg/t to between 0.19 kg/t and 0.32 kg/t, with gold recovery ranging from 93.4% to 97.8%. This condition has been retained in the conceptual design basis;
- A gravity-recoverable gold fraction of 43.9% to 61.4% is present in all three deposits at the design grind size, but cyanidation of the gravity tailings returned an equivalent overall recovery, so no recovery credit has been taken for gravity separation;
- Cyanide destruction by the SO_2 /air process was effective but required a higher than usual SO_2 dosage to reduce the CNwad below 2 ppm. This is probably a result of the presence of a significant pyrrhotite content. A dosage of 8 g SO_2 /gCNwad is then required instead of a value of 5 that usually work for mill feed that contains less sulphide.

A recovery equation based on grind size, feed gold grade and feed sulphur grade was established from the leach results and is used to project gold recovery over the mine plan. The testwork provides an adequate basis for projecting the metallurgical performance of cyanide leach processing and estimating the process operating cost at the preliminary economic assessment level. Further testwork will be required to advance the project beyond this level.

1.9 Mineral Resource Estimate

The mineral resource estimate presented in this Report covers the Cadillac Project.

Leapfrog Geo™ and Edge™ v.2025.1.1 (“Leapfrog”) was used to update the mineralized zones and to generate the drill hole intercepts for each solid. Leapfrog was used for the compositing, 3D block modelling, grade interpolation, and classification. Statistical studies were conducted using Excel, Leapfrog, and Supervisor. The pit optimization to develop the mineral resource-constraining pit shells was done using the pseudoflow algorithm in Deswik software. The stope optimization to develop the underground mineral resource was done using Deswik.SO software. The methodology for the mineral resource estimation involved the following steps:

- Database verification and validation;
- 3D modelling update of the mineralized zones;
- Drill hole intercepts and composite generation;



- Basic statistics;
- Capping;
- Geostatistical analysis including variography;
- Block modelling and grade interpolation;
- Block model validation;
- Mineral resource classification;
- Cut-off grade calculation;
- Pit shell optimization;
- DSO optimization;
- Preparation of the mineral resource statement.

The 2025 MRE is constrained within a pit shell developed and DSO shapes using appropriate cut-off grades. Table 1-1 presents the results of the MRE.

Table 1-1: Cadillac Project Mineral Resource Estimate

	Open Pit Resources			Underground Resources			Total		
	Cut-off Grade = 0.30g/t Au			Cut-off Grade = 1.00g/t Au					
	Tonnes (t)	Grade (Au g/t)	Gold (oz)	Tonnes (t)	Grade (Au g/t)	Gold (oz)	Tonnes (t)	Grade (Au g/t)	Gold (oz)
Measured	1,770,000	2.16	123,300	4,210,000	2.80	379,300	5,988,000	2.61	502,600
Indicated	1,730,000	1.52	84,500	2,240,000	2.51	180,600	3,965,000	2.08	265,200
M&I	3,500,000	1.84	207,800	6,450,000	2.70	559,900	9,953,000	2.40	767,800
Inferred	4,740,000	1.13	172,600	30,450,000	2.29	2,244,200	35,185,000	2.14	2,416,900

Notes to Table 1-1:

- The independent qualified person for the MRE, as defined by National Instrument ("NI") 43-101 guidelines, is Pierre Luc Richard, P.Geo., of PLR Resources Inc. with contributions from Stephen Coates, P.Eng., of Evomine for cut-off values, open pit optimization solids and underground optimization solids, and Florent Baril, P.Eng., from Bumigeme Inc., for metallurgical parameters. The effective date of the MRE is November 11, 2025.
- These Mineral Resources are not mineral reserves as they have no demonstrated economic viability. No economic evaluation of these Mineral Resource has been produced. The quantity and grade of reported Inferred Resources in this MRE are uncertain in nature and there has been insufficient drilling to define these Inferred Resources as Indicated. However, it is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated category with continued drilling.
- The Qualified Persons are not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, marketing or other relevant issues that could materially affect the Mineral Resource Estimate.
- Calculations used metric units (meters (m), tonnes (t), and g/t). Metal contents in the above table are presented in troy ounces (metric tonne x grade / 31.103475). Values were rounded, and any discrepancies in total amounts are due to rounding errors.
- The Mineral Resource estimate follows the November 29, 2019, CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines.



6. Resources are presented as undiluted and in situ for the open-pit scenario and include internal dilution for the underground scenario and are considered to have reasonable prospects for economic extraction. The constraining pit shell was developed using overall pit slopes of 50 degrees. The pit optimization to develop the mineral resource-constraining pit shells was done using the pseudoflow algorithm in Deswik software. The stope optimization to develop the underground mineral resource was done using Deswik.SO software.
7. The MRE wireframe was prepared using Leapfrog Edge v.2025.1.1 and is based on 4,477 drill holes, totalling 535,360 meters drilled and 167,978 assays. The cut-off date for the drill hole database was February 17, 2025.
8. Composites of 1.0 metre were created inside the mineralization domains. High-grade capping was done on the composited assay data. Based on individual statistical study for each zone, composites were capped between 5.0 g/t Au and 110.0 g/t Au for the high-grade zones, and between 1.0 g/t Au and 5.0 g/t Au for the low-grade zones.
9. Pit constrained Mineral Resources for the base case are reported at a cut-off grade of 0.30 g/t Au; DSO-constrained Mineral Resources for the base case are reported at a cut-off grade of 1.00 g/t Au and include internal dilution (must-take). The cut-off grades will be re-evaluated in light of future prevailing market conditions and costs.
10. Specific gravity values were estimated using data available in the drill hole database. Density values between 2.80 and 2.88 were applied to the model for hard rock and 2.00 for overburden.
11. Grade model resource estimation was calculated from drill hole data using an Ordinary Kriging interpolation method in a sub-blocked model using blocks measuring 5 m x 5 m x 5 m in size and sub-blocks down to 0.625m x 0.625m x 0.625m. Ordinary kriging (OK), inverse square distance (ID2), Nearest neighbour (NN) interpolation methods were tested, resulting in no material difference in the Mineral Resource Estimates.
12. The Measured, Indicated and Inferred Mineral Resource categories are constrained to areas where drill spacing is less than 12.5m, 25m and 100 metres respectively and show reasonable geological and grade continuity. An additional requirement for the Measured category is the close proximity of underground infrastructure. Cookie cutters were used to define categories based on the above parameters. Based on historical mining and geological knowledge of the deposit, drill spacing was increased up to 140m in the shoot direction to define inferred resources for some zones.

1.10 Exploration potential

After reviewing all pertinent information, including the MRE, the QP concluded the following:

- The potential is high for adding underground mineral resources to the Cadillac Project by extending 3D modelling at depth and laterally.
- The potential to upgrade Inferred Mineral Resources to the Indicated category with additional drilling is high.
- The exploration potential remains high at the Project scale, justifying further geological compilation and continuing exploration target generation programs.

A Conceptual Exploration Target, with underground potential, was identified during the preparation of the MRE. This conceptual Exploration Target is integrated into the model used for the MRE, with the aim of facilitating future targeting and drill hole planning.

The assessment of the target for further exploration was completed by PLR Resources Inc. with contribution from Evomine. The estimation of the potential quantity and grade of the exploration



target was based on the same drill hole database used for the Mineral Resource Estimate. With the available drilling information, PLR developed conceptual gold mineralization volumes. The original core samples were composited, and the composited gold assays were capped (similarly to the Mineral Resource Estimate) after evaluating the statistical distributions on probability plots. The gold values were interpolated into a three-dimensional block model using Ordinary Kriging. To estimate a tonnage, PLR used the same specific gravity values used for the Mineral Resource Estimate.

DSO stopes were run to constrain the Exploration Target.

The Conceptual Exploration Target is estimated to be of 8 to 12 million tonnes of mineralization grading between 2.2 to 2.8 g/t Au, representing 600,000 to 1,100,000 ounces of gold.

Please note the following disclosure warnings in respect to this exploration target:

- An Exploration Target is not a National Instrument 43-101 compliant resource or reserve.
- The Exploration Target is confirmed only as a target for further exploration.
- Potential quantity and grades are conceptual in nature only.
- There has not been sufficient drilling to define any mineral resource on this Exploration Target; drilling intercepts crosscut the Exploration Target but drill spacing is too scarce to classify these blocks as Inferred Mineral Resources.
- There is no certainty that further drilling will result in the target being delineated as a mineral resource.

1.11 Mining Methods

The Cadillac Project is planned as a mechanized underground mining operation using longitudinal longhole open stoping (LHOS), with a combination of cemented rockfill, uncemented rockfill and paste fill for stope backfilling. This mining method was selected based on the geometry of the mineralized zones, historical mining experience at the historic Chimo Mine, and the anticipated ground conditions. It provides a mechanized, non-entry mining approach intended to support productive and cost-effective extraction.

The mine plan encompasses six mineralized domains: Main–Chimo, Main–East, Nordeau, Rapid, Contact and Portal. These domains will be accessed through five mine portals on surface and dedicated ramp systems, with mining areas developed and extracted sequentially from the bottom up for each mining sill. The planned stopes generally have a nominal vertical height of 25 m and a strike length of 17.5 m, with a minimum mining width of 2.0 m. Mining dilution allowances are based



on equivalent linear overbreak/slough (ELOS) factors of 0.5 m for the hanging wall and 0.2 m for the footwall. A 30 m buffer is maintained around historical mine workings to mitigate potential ground stability risks.

The underground mine schedule includes a 24-month pre-production period dedicated to portal construction, ramp and lateral development, ventilation infrastructure, and initial stope preparation. Following a three-month production ramp-up, the operation is scheduled to achieve a nominal production rate of 3,000 tonnes per day (tpd) during the first four years of commercial operations, increasing to 4,300 tpd in Year 5 and remaining at this rate for most of the remaining mine life. The overall project schedule comprises 18.2 years, including pre-production and 16.2 years of commercial operations. Approximately 23.1 million tonnes of mineralized material are scheduled to be mined at an average diluted gold grade of 2.28 g/t.

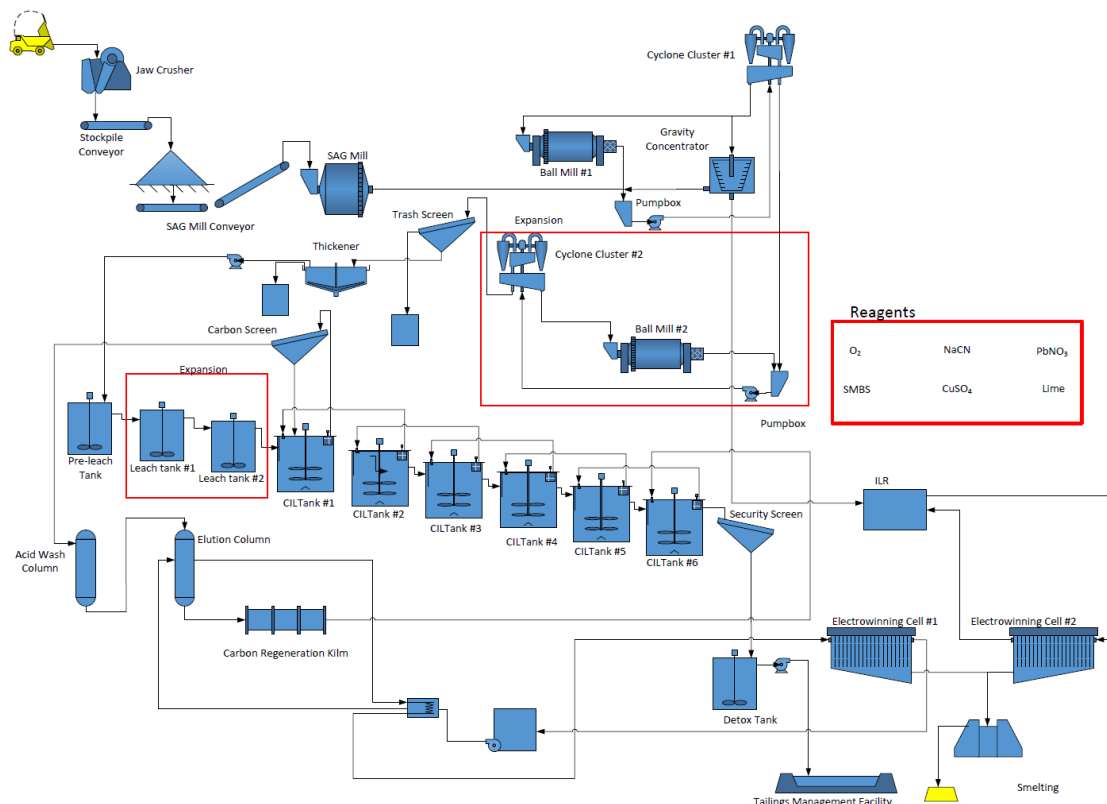
The mine design requires approximately 234 km of lateral underground development and 6.4 km of vertical development. Mining operations during both the pre-production period and the production period will be owner-executed, using a mechanized fleet that includes production and development drills, load-haul-dump (LHD) units, underground haul trucks, rockbolters and auxiliary equipment. Mineralized material will be hauled directly to surface and stockpiled near the portals before delivery to the processing facility. Development waste rock will be transported to designated surface waste rock management facilities or used underground as backfill, where appropriate.

Backfilling is an integral component of the mining sequence and ground stability strategy. The use of paste fill is expected to allow a portion of the processing tailings to be returned underground, reducing the volume of tailings requiring surface storage and potentially limiting additional waste rock requirements. Ventilation will be provided through a staged system comprising temporary ventilation during initial development and a permanent network of fresh-air raises, surface fans and booster fans. Underground services will also include dewatering, compressed air, communications, fuel distribution and emergency infrastructure.

The mining design and production schedule are preliminary and rely on historic geomechanical information, empirical design methods and experience from comparable operations. No new site-specific geomechanical or hydrogeological investigations were completed for this assessment. Further geotechnical drilling, rock mass characterization, hydrogeological studies, and validation of stope stability, ground support, crown pillar dimensions and backfill strength will therefore be required during subsequent engineering studies to confirm and optimize the mine design.

The process plant is designed around conventional unit operations. It will process 3,000 tonnes per day during Phase 1, increasing to 4,300 tonnes per day during Phase 2. The crushing circuit is designed for 65% availability, while the milling circuit is designed for 92% availability, equivalent to 8,059 operating hours per year. The plant will operate 24 hours per day, 365 days per year, and will produce doré bars. During pre-production and the first year of operation, mine production is expected to be processed at a custom mill. Treatment costs for this period are based on a realistic custom-milling fee and transportation to the facility. Gold recovery is expected to be lower because the custom mill grind size (P80) will likely be coarser than 50 µm.

Figure 1-1 : Process Flow Diagram



crusher will reduce the mill feed before it is conveyed to the crushed material stockpile. The crushed mineralized material will then be ground in a SAG mill followed by a ball mill operating in closed circuit with a hydrocyclone cluster. During Phase 2, a second ball mill and hydrocyclone cluster will be added in closed circuit to maintain a product P80 grind size of 50 μm at the increased throughput. A portion of the hydrocyclone underflow will be diverted to a centrifugal gravity concentrator to recover liberated gold. The gravity concentrate will be treated in an intensive cyanide leach reactor, and the resulting pregnant solution will be directed to the electrowinning circuit in the gold room. No incremental gold recovery is credited to the gravity circuit; it is included to limit the recirculation and accumulation of liberated gold within the grinding circuit. The hydrocyclone overflow, at a product P80 grind size of 50 μm , will be directed sequentially to a thickener, a pre-aeration tank and the carbon-in-leach circuit.

Gold will be leached in the carbon-in-leach circuit and adsorbed onto activated carbon. The loaded carbon will be eluted, and gold will be recovered from the pregnant eluate by electrowinning in the gold room. The electrowon gold sludge will be dried and smelted to produce doré bars. The eluted carbon will be thermally reactivated in a carbon regeneration kiln before being returned to the CIL circuit. The CIL tailings will undergo cyanide destruction before being pumped to the tailings management facility. During Phase 2, two additional leach tanks will be installed between the pre-aeration tank and the CIL circuit to maintain the design cyanide-leaching residence time at the increased throughput.

The process plant power demand is estimated at 10.1 MW during Phase 1 and 12.5 MW during Phase 2, with corresponding annual energy consumption of 63,600 MWh/y and 79,500 MWh/y, respectively. Raw water will be pumped to a storage tank that will supply the process plant and reagent preparation area. Process-water demand will be met primarily by reclaim water from the tailings management facility and thickener overflow. The principal reagents will comprise pebble lime, sodium cyanide, sodium hydroxide, lead nitrate, copper sulphate pentahydrate, hydrochloric acid, sodium metabisulphite, activated carbon and flocculant.

1.13 Project Infrastructure

The Cadillac Project infrastructure has been conceptually designed to support underground mining operations over the 16.2-year production period, including the planned production ramp-up from 3,000 tonnes per day (tpd) to 4,300 tpd. The infrastructure scope includes site access and internal roads, mine portals and services, administration and maintenance buildings, material stockpiles,



water management facilities, and electrical and communications systems. The processing strategy considers a staged approach, with off-site toll milling planned during the initial operating period and an on-site processing plant and associated tailings storage facilities included in the overall project infrastructure concept.

The site layout is organized into functional zones to promote operational efficiency, personnel safety, and environmental management. Five portal areas provide access to the underground mining areas, with adjacent run-of-mine (ROM) mineralized material stockpiles and waste rock management facilities. This arrangement is intended to facilitate material handling and allow waste rock to be transported underground for backfill. Centralized surface facilities include the mine dry, administrative offices, mobile equipment maintenance building, fuel station, laydown areas, cement storage silos, and paste backfill plant. No on-site accommodation camp is planned, based on the assumption that employees will commute from nearby communities, most notably Val-d'Or. Site access is provided by Route 117 and existing gravel and forestry roads, which will be upgraded and maintained to accommodate project traffic and heavy equipment. New internal roads will connect the main operational facilities. Underground services will include temporary ventilation systems during early development, followed by permanent ventilation raises and primary fans, as well as compressed air, dewatering, electrical distribution, and communications infrastructure. Propane storage and distribution systems will provide fuel for ventilation air heating. A private LTE communications network is proposed to support voice and data communications, personnel and equipment tracking, and underground connectivity.

Water management infrastructure will collect contact water from surface runoff and underground operations through drainage ditches and collection basins. Collected water will be stored and treated before discharge in accordance with applicable regulatory requirements. Electrical power is expected to be supplied by the regional grid through a 120 kV powerline along the right-of-way of an existing 25 kV powerline. A main surface substation will distribute power to surface facilities and underground operations, with underground feeders conceptually rated at 13.8 kV.

The conceptual designs for surface facilities also include five overburden management facilities, waste rock management facilities, and contact water retention basins. Preliminary geotechnical design assumptions have been adopted for stockpile slopes, basin slopes, and building foundations; however, no site-specific geotechnical investigations have yet been completed. These assumptions will require validation through geotechnical drilling, laboratory testing, and stability analyses during subsequent studies. Further engineering will also be required to finalize facility

layouts and dimensions, electrical loads, ventilation and backfill systems, water treatment capacities, and infrastructure specifications.

During pre-production and the initial year of the commercial production period, mineralized material will be transported to a third-party mill under a proposed toll-milling arrangement. Preliminary supporting infrastructure at the toll mill site includes a designated stockpile area, weigh station, fencing, and security systems. The final selection of the toll mill, transportation logistics, and development of the on-site processing facilities will be refined during subsequent engineering phases.

1.14 Market Studies and Contracts

Gold is a globally traded, highly liquid commodity, with established international markets and extensive refining capacity in Canada and worldwide. No project-specific market studies have been commissioned for the sale of gold doré or concentrates from the Cadillac Project. Based on a review of macroeconomic trends, historical prices, and consensus forecasts, the Qualified Person considers the gold price assumptions adopted for this updated Preliminary Economic Assessment (PEA) reasonable and appropriate. No long-term offtake agreements, hedging arrangements, streaming agreements, or fixed-price sales contracts are currently in place. Gold doré is expected to be sold to reputable refiners or smelters on competitive commercial terms.

The updated PEA uses a base-case gold price of US\$3,600 per troy ounce, compared with a three-year trailing monthly average of US\$3,177 per ounce as of August 31, 2026. The financial model also assumes an exchange rate of CAD 1.38 per US\$1.00. Since gold revenues are denominated in US dollars while most project costs are incurred in Canadian dollars, exchange-rate fluctuations represent a potential source of financial risk; sensitivity analyses have been conducted to assess their impact on project economics.

The project development strategy includes toll milling at an existing third-party facility during pre-production and the first year of commercial operations, with the dedicated on-site processing plant scheduled to begin operating at the start of Year 2. As of the effective date of the report, Cartier Resources Inc. had not received any non-binding letters of intent for toll milling. The processing assumptions are therefore based on publicly available commercial terms and regional practices in the Abitibi greenstone belt.

Insurance, transportation, and gold refining contracts will be negotiated and finalized during construction. The updated PEA estimates combined insurance, transportation, and refining costs at



US\$5 per troy ounce of gold.

1.15 Environmental Studies, Permitting and Social or Community Impact

The construction, operation, and closure of the mine are subjected to several laws and regulations at the federal, provincial, and municipal levels. The Cadillac project is not subject to the federal environmental assessment procedure with a production capacity below 5,000 t/d. At the provincial level, all work required for the operation of a new mine are subject to the impact assessment procedure. The Cadillac project will be subject to the procedure for impact assessment of development projects in meridional Quebec with an eventual review by the “Bureau d’audiences publiques sur l’environnement” (BAPE). It is recommended to start baseline studies and consultations with stakeholders early in the development of the project.

Moreover, the Mining Act provides a framework for the mining lease, rehabilitation and restoration plan, and financial guarantee. The closure plan must be approved by the Ministry of Natural Resources and Forests (MRNF) before the issue of the mining lease, and a financial guarantee to fully implement the closure plan must be provided in the first two years following its approval. Estimated closure costs of approximately C\$40 million have been benchmarked against recent mining projects in similar jurisdictions.

The environmental permitting process requires an understanding of the physical (surface water, groundwater, air, noise, etc.), biological (fauna and flora) and human (land and resource uses, archaeology, landscape, etc.) environments, and include an evaluation of the potential impacts of the project and proposed mitigation measures. Environmental studies and geochemical characterization have been started by Stantec in 2026. Major components of the physical, biological and human environments have been described in an initial desk review study. Environmental baseline studies will start in the next phase of the project, and the geochemical characterization of waste rock and mineralized material is currently ongoing.

As part of the development of the Cadillac project, several information and consultation meetings with stakeholders and First Nation authorities are required. An engagement process will be implemented by Cartier to meet various stakeholders concerned by the Cadillac project, including municipal and regional authorities, development economic organizations, environmental groups, nearby cottagers and land users. Meetings will also be held with representatives of the Anishnabe



Nation Council of Lac-Simon.

1.16 Capital and Operating Costs

Life-of-mine project capital costs are estimated to total \$1,685.1 million gross of pre-production revenue (\$1,654.9 million net of pre-production revenue), split in the following the following categories:

- Initial Capital Costs – This category includes all costs to develop the property and does not include a process plant, as the project starts with a toll mining operation for the first year of production. Initial capital costs total \$306.0 million (including \$37.9 million for contingency but gross of \$30.2 million in pre-production revenue), which will be expended over a 24-month period of construction, pre-production and commissioning.
- Growth capital costs – This category includes all costs related to the construction of process plant expended during the first year of production, as well as the expansion of this plant during the fourth year of production. Growth capital costs are estimated to be \$277.1 million, inclusive of a \$37.8 million contingency.
- Sustaining Capital Costs – This category includes all costs related to the acquisition, replacement, or major overhaul of assets required to sustain operations, as well as underground development and infrastructure during the production phase of the Project. Sustaining capital costs are estimated to be \$1,062.0 million, inclusive of a \$138.5 million contingency.
- Working Capital – This category consists of costs related to building up an inventory. It is planned to maintain a supplies inventory equivalent to 45 days of operating expenses, as well as a finished goods inventory equivalent to 7 days of revenue. Additionally, it is planned to pay the various operational suppliers within 30 days, and 7 days of accounts receivable are considered. Though the total net changes in working capital over the life of mine is null, the maximum amount of working capital for the Project is \$34.9 million.
- Closure Costs – This category includes all costs related to the closure, reclamation, and ongoing monitoring of the mine after operations. Closure costs are estimated at \$40.0 million.

The total capital costs are summarized in Table 1-2.



Table 1-2 : Capital Costs Summary

Cost Area	Initial Capital Costs (\$M)	Growth Capital Costs (\$M)	Sustaining Capital Costs (\$M)	Total Capital Costs (\$M)
Infrastructure	40.9	-	52.5	93.4
Electrical and communication	27.9	-	8.1	36.0
Waste and water management	9.2	-	58.7	68.0
Mine	91.3	-	720.2	811.5
Process plant	9.0	230.2	-	239.2
Indirect costs	29.8	9.0	84.0	122.7
Contingency	37.9	37.8	138.5	214.2
Capitalized operating costs	60.0	-	-	60.0
Closure	-	-	40.0	40.0
Total	306.0	277.1	1,102.0	1,685.1
Pre-production revenue	-30.2	-	-	-30.2
Total (net of pre-production revenue)	275.8	27..1	1,102.0	1,654.9

Operating costs are summarized in Table 1-3. The operating costs include mining, processing, waste and water management and general services and administrative (“G&A”) costs. The average life of mine operating cost is US\$1,481/oz of gold payable, including selling and royalty costs. The average life of mine all-in sustaining cost (“AISC”) is US\$2,107/oz of gold payable.



Table 1-3 : Operating Costs Summary

Cost Area	Total Cost (\$ M)	Unit Cost		Unit Cost (US\$/oz payable)
Underground mining	2,193.7	95.1	\$/tonne processed	996.0
Off-site transport	15.1	0.7	\$/tonne processed	6.8
Off-site processing	61.8	2.7	\$/tonne processed	28.1
On-site processing	610.5	26.5	\$/tonne processed	277.2
Waste and water management	34.6	1.5	\$/tonne processed	15.7
General and administrative	232.5	10.1	\$/tonne processed	105.6
Subtotal	3,148.2	136.4	\$/tonne processed	1,429.4
Selling costs	11.1	0.5	\$/tonne processed	5.0
Royalty (1.5%)	101.7	4.4	\$/tonne processed	46.2
Total	3,260.9	141.3	\$/tonne processed	1,480.6
Growth capital	277.1	12.0	\$/tonne processed	125.8
Sustaining capital (incl. closure costs)	1,102.0	47.8	\$/tonne processed	500.4
All-in Sustaining Costs ("AISC")	4,640.0	201.1	\$/tonne processed	2,107.0

The peak total workforce for the Cadillac Project, which includes personnel for the mining operation, processing, and general and administrative activities is estimated at 515.

1.17 Economic Analysis

The economic and financial evaluation presented in this Technical Report utilizes a discounted cash flow method, both on a pre-tax and after-tax basis. The gold price used in the evaluation was US\$3,600 per troy ounce, as discussed in Item 19 of this Technical Report. The financial model provides results in terms of NPV, payback period, and IRR for the Cadillac project. The economic analysis is conducted in real terms, without considering inflation factors, using Q3 2026 Canadian dollars. The analysis does not take into account project financing.

The economic model estimates cash flows on an annual basis for the life of the Cadillac project, based on the level of engineering and design appropriate for a PEA.

Cash flow projections for the life of the mine are based on sales revenue, operating costs, capital expenditures and other cost estimates. Capital costs are estimated in 5 categories: initial capital,



growth capital, sustaining capital, closure and working capital. Operating cost estimates include labour, reagents, maintenance, supplies, services, fuel, and power. Other costs, such as royalties, depreciation, and taxes, are estimated based on the current project and processing plans.

The economic results are calculated from the start of the initial capital expenditures.

A summary of the Project economic results is presented in Table 1-4. The total after-tax undiscounted cash flow over the Project life is \$1,867 million and after-tax NPV5% is \$1,001 million. The after-tax Project cash flow results in a 4.3-year payback period from the commencement of commercial operations with an after-tax IRR of 26.6%.

Table 1-4: Economic analysis highlights

	Unit	Base case	Spot case
Gold price	US\$/oz	3,600	4,300
Exchange rate	\$/US\$	1.38	1.38
Pre-tax free cash flow	\$ million	3,013	4,541
Pre-tax NPV5%	\$ million	1,679	2,630
Pre-tax IRR	%	35.5	51.0
Pre-tax payback period	Years	3.0	2.1
After-tax free cash flow	\$ million	1,867	2,772
After-tax NPV5%	\$ million	1,001	1,565
After-tax IRR	%	26.6	37.3
After-tax payback period	years	4.3	2.7

1.18 Interpretation and Conclusions

This Technical Report is prepared in accordance with the guidelines of the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101") and Form 43-101F1. The objective of this Report and the updated PEA is the evaluation of the potential technical feasibility and economic viability of the Project, notably the development of an underground mine thereat, including supporting infrastructure. This Report confirms the potential viability of the Project based on an underground mining operation that generates, on an after-tax basis, a NPV5% of \$1,001 million, a 4.3-year payback period and an IRR of 26.6%.

The QPs consider the updated PEA to be reliable, thorough, based on quality data, reasonable hypotheses, and parameters compliant with NI 43-101 requirements and CIM Definition Standards.



1.19 Recommendations

The results of the updated Preliminary Economic Assessment (“PEA”) indicate that the Project has the potential to be economically attractive under the assumptions and parameters considered in this study. The economic analysis presents positive valuation metrics and supports advancing the Project to the next stage of technical evaluation. However, the assessment is preliminary in nature, and significant technical investigations, engineering studies, and further evaluation of key risks are required to improve the level of confidence in the Project's development concept and economic performance.

It is therefore recommended that the Project advance to a structured program of technical work aimed at validating the principal assumptions of the updated PEA, reducing key technical and economic uncertainties, and establishing a sound basis for a future Pre-Feasibility Study (“PFS”). The recommended work program should include, but not necessarily be limited to, the following activities.

Mineral Resources and Drilling

- Undertake additional infill drilling within the areas that contribute most significantly to the proposed mine plan, with the objective of improving confidence in the Mineral Resource Estimate and supporting the potential conversion of Inferred Mineral Resources to Indicated Mineral Resources, where justified.
- Conduct additional geological and structural investigations to refine the understanding of mineralization geometry, continuity, and relevant geological controls.
- Update the geological model and Mineral Resource estimate as warranted by the results of the additional drilling and supporting investigations.

Geotechnical and Mine Design

- Conduct a geotechnical investigation program, including oriented core drilling, rock mass characterization, structural data collection, and laboratory testing, to support the development of geotechnical domains and appropriate rock mass design parameters.
- Complete preliminary underground mine geotechnical analyses to establish appropriate excavation dimensions, ground support requirements, stope design parameters, and mine stability criteria.
- Refine the mine design, production schedule, development requirements, and mining method assumptions based on the updated geological model and geotechnical findings.

- Evaluate the implications of the revised mine design on mining productivity, dilution, mineralized material loss, mine sequencing, and operating costs.

Hydrogeology and Water Management

- Undertake a hydrogeological investigation program to characterize groundwater conditions, hydraulic properties, potential groundwater inflows, and the principal hydrogeological controls affecting mine development and operations.
- Develop a preliminary groundwater model, as appropriate, to support estimates of mine inflows, dewatering requirements, and potential effects on surrounding groundwater and surface water systems.
- Refine the mine water management concept, including water collection, pumping, treatment, storage, discharge, and/or reinjection requirements, as applicable to the Project.

Metallurgy and Mineral Processing

- Complete additional metallurgical testwork to confirm and refine the assumed metallurgical recoveries and assess variability in feed characteristics across the proposed mine plan.
- Investigate potential impacts of mineralogical variability, deleterious elements, and processing conditions on concentrate quality, recoveries, and operating costs.
- Refine the process flowsheet, plant design criteria, and associated capital and operating cost estimates based on the results of the additional testwork.
- Engage in discussions with third party mill operators to establish a toll milling agreement.

Infrastructure, Waste Management, and Environmental Studies

- Advance the engineering of the principal site infrastructure, including access roads, electrical supply, water management facilities, and other facilities required to support the proposed mining and processing operations.
- Complete preliminary geotechnical and geochemical characterization of waste rock and other relevant mine materials to support the design of waste management facilities and the assessment of potential environmental liabilities.
- Advance the conceptual designs for tailings and waste rock management facilities, including site selection, capacity requirements, foundation conditions, and water management considerations, as applicable.
- Initiate or continue baseline environmental studies and permitting-related investigations to identify potential environmental constraints, assess the principal project impacts, and

support future regulatory submissions.

- Engage with relevant stakeholders and First Nation communities, as appropriate, to identify potential concerns and incorporate applicable considerations into the Project development plan.

Engineering and Economic Evaluation

- Advance engineering and design activities to an appropriate level of detail for a PFS, incorporating the results of the recommended technical investigations.
- Update capital and operating cost estimates using refined engineering quantities, equipment requirements, productivity assumptions, and current pricing information.
- Update the production schedule and economic model to reflect the revised technical assumptions, cost estimates, metallurgical parameters, and any material changes to the proposed development concept.
- Conduct sensitivity analyses and risk assessments to identify the principal factors affecting Project economics and determine which uncertainties warrant further investigation.
- Evaluate opportunities to optimize the mine plan, infrastructure configuration, processing strategy, and overall development approach, with emphasis on improving economic performance while managing technical and execution risks.

The recommended work should be prioritized according to its potential to affect the Project's economic performance and the level of uncertainty associated with the relevant assumptions. In particular, the drilling, geotechnical, hydrogeological, and metallurgical programs should be designed to address the uncertainties that have the greatest potential to influence the mine plan, capital and operating costs, and economic valuation.

Subject to the results of these investigations and confirmation that the principal technical and economic assumptions remain supportable, the Project should then be advanced to a Pre-Feasibility Study. The PFS should establish a more detailed and integrated mine development plan, infrastructure design, production schedule, cost estimate, and economic evaluation, while further assessing the principal technical, environmental, permitting, and execution risks. Table 1-5 highlights the proposed budget for the work program.



Table 1-5: Work program budget

Item	Budget
Expansion and definition drilling	\$31,250,000
Exploration drilling	\$3,750,000
Geotechnical investigation and analysis	\$750,000
Hydrogeological investigation and analysis	\$750,000
Metallurgical and mineralogical testwork	\$750,000
Geochemical characterization of material	\$250,000
Environmental baseline	\$2,000,000
Stakeholder engagement	\$100,000
Pre-Feasibility Study including updated Mineral Resource Estimate	\$2,000,000
Total	\$41,600,000

Although the updated PEA presents positive preliminary economic results, additional work is required before the Project's technical feasibility and economic viability can be established with greater confidence. The results of the updated PEA should therefore be considered an initial assessment of the Project's potential rather than a definitive basis for a development decision.



2 INTRODUCTION

Cartier Resources Inc. (“Cartier”) mandated Evomine Consulting inc. as lead consultant, along with other engineering consultants, to prepare an updated Preliminary Economic Assessment (“PEA”) under the supervision of the QPs for the Cadillac Project (the “Project”), located approximately 35 km east-southeast of the city of Val-d’Or, in the Abitibi-Témiscamingue region of Québec.

Cartier Resources Inc. (“Cartier”) is listed on the TSX Venture Exchange under the symbol “ECR”, and is also quoted on the OTCQB (“ECRFF”) and the Frankfurt Stock Exchange (“6CA”). Its head office is in Val-d’Or, Québec. The issuer, Cartier Resources Inc., is a Canadian mineral exploration company focused on the exploration and development of gold properties in the Abitibi Greenstone Belt. Cartier holds a 100% interest in the Cadillac Project, which extends over approximately 15 kilometres along the Larder Lake–Cadillac Fault Zone and includes the past-producing Chimo Mine, which closed in 1997.

This Technical Report is prepared in compliance with the Canadian Securities Administrators’ National Instrument 43-101 (“NI 43-101”) and Form 43-101F1. The objective of this Report is the evaluation of the potential technical and economic viability of the Project, notably the development of five underground mining areas (Main-Chimo, Main-East, Nordeau/Contact, Rapid and Portal) including supporting infrastructure. This Report provides operating and capital cost estimates and an economic analysis of the Project.

The updated PEA restates the same Mineral Resource Estimate that was issued on January 27, 2026 with an effective date of November 11, 2025, and on the technical and economic assumptions, qualifications and limitations described in Sections 14, 15, 16, 17, 18, 19, 21 and 22 of this Report. These assumptions include a gold price of US\$ 3,600/oz, an exchange rate of 1.38 C\$/US\$, 90.5% toll milling gold metallurgical recovery, an average of 94.8% gold recovery from on-site milling, and mining and processing rates of 3,000 tonnes per day during Phase 1, increasing to 4,300 tonnes per day during Phase 2. The Mineral Resource statement reported herein was prepared in conformity with the generally accepted CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines. The updated PEA presented in this Report is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the updated PEA will be realized. Mineral Resources that are not Mineral



Reserves do not have demonstrated economic viability. No Mineral Reserve has been declared for the Project.

The effective date of this Technical Report is August 31, 2026, and the issue date is October 30, 2026.

2.1 Sources of Information and Validation

Unless otherwise stated, all the information and data contained in the Report or used in its preparation has been provided by up to August 31, 2026. The QPs have no reason to doubt the reliability of the information provided.

Sources of information include:

- Discussions with Cartier personnel;
- Inspection of the Cadillac Project area, including drill collars, drill core, and ground conditions;
- Drilling database received;
- Geological interpretations and exploration data;
- Metallurgical test work results;
- Technical and scientific reports by external consultants;
- Costing information received from suppliers and service providers;
- All documents referenced in Section 27.

The independent qualified persons have reviewed the following with respect to the Cadillac Property:

- Mining titles and their status recorded in GESTIM (the Government of Québec's online EER management system).
- Agreements and technical data supplied by the issuer (or its agents).
- Public sources of relevant technical information available through SIGÉOM (the Government of Québec's online warehouse for assessment work).
- Issuer's filings on SEDAR (e.g., press releases and Management's Discussion & Analysis reports).

2.2 Qualified Persons

The Qualified Persons for this Technical Report are the following:

- | | |
|------------------------------|-------------------------|
| ➤ Alexandre Burelle, P. Eng. | Evomine Consulting Inc. |
| ➤ Stephen Coates, P. Eng. | Evomine Consulting Inc. |
| ➤ Maude Levesque-Michaud | Geodoz Conseil Inc. |
| ➤ Pierre-Luc Richard, P.Geo. | PLR Resources Inc. |



> Pierre Roy, P. Eng.

Soutex Inc.

Table 2-1 highlights the qualified persons, the sections for which they are responsible and the details of their personal inspection of the property.

Table 2-1 : List of Qualified Person

Qualified Person	Firm	Item Responsibility	Personal Inspection
Alexandre Burelle, P. Eng.	Evomine Consulting Inc.	1.11, 1.13, 1.16, 1.17, 15, 16, 18, 21 (except 21.3.6, 21.4.1, 21.5.1, 21.7.2 & 21.7.5.2), 22, 25.7, 25.9, 25.10, 27	September 18, 2025
Stephen Coates, P. Eng.	Evomine Consulting Inc.	1.1, 1.2, 1.3, 1.14, 1.18, 1.19, 2, 3, 14.14, 19, 25.1, 25.11, 25.12, 26, 27	September 18, 2025
Maude Levesque-Michaud, P.Eng., M.Sc.A.	Geodoz Conseil Inc.	1.15, 1.16, 20, 21.5.1, 25.8, 27	None; QP judged that it was not necessary for site visit at this level of project development
Pierre-Luc Richard, P.Geo	PLR Resources Inc.	1.4, 1.5, 1.6, 1.7, 1.9, 1.10, 4, 5, 6, 7, 8, 9, 10, 11, 12, 14 (except 14.14), 23, 24, 25.2, 25.3, 25.4, 25.5, 27	May 26, 2025
Pierre Roy, P. Eng.	Soutex Inc.	1.8, 1.12, 1.16, 13, 17, 21.3.6, 21.4.1, 21.7.2, 21.7.5.2, 25.6, 27	None; QP judged that it was not necessary for site visit at this level of project development

2.3 Units and Currency

The units of measure presented in this Technical Report, unless noted otherwise, are in the metric system. Currency is expressed in Canadian dollars (“CAD”), unless stated otherwise. Costs are based on third quarter (“Q3”) 2026 dollars. A list of the main abbreviations and terms used throughout this Technical Report is presented in Table 2-2.

Table 2-2 : Abbreviations and acronyms

Abbreviation or Symbol	Description
\$ or CAD	Canadian dollar
%	Percent
% w/w	Percentage in weight by weight
°C	Degree Celsius
µm	micrometer
A x b	Grindability factor for mill design
AA	Atomic-Absorption
ABA	Acid-Base Accounting



Abbreviation or Symbol	Description
Ag	Silver
AP	Acidification Potential
ATV	All-Terrain Vehicle
Au	Gold
BAPE	<i>Bureau d'audiences publiques sur l'environnement</i>
BHEM	Borehole Electromagnetic
BV	Bed Volume
CA\$, CAD, \$	Canadian Dollar
CAPEX	Capital Expenditures
Ccarb	Carbonate carbon
CCREM	Canadian Council of Resource and Environment Ministers
CCTV	Closed-Circuit Television camera system
CDPNQ	<i>Centre de données sur le patrimoine naturel du Québec</i>
CIL	Carbon in Leach
cm	Centimeter
CMHPQ	<i>Cartographie des milieux humides potentiels du Québec</i>
CNG	Cree National Government
CN _{WAD}	Weak Acid Dissociable Cyanide
COFEX	Federal Review Committee
COG	Cut-Off Grade
COMEX	Environmental and Social Impact Review Committee
CPTAQ	<i>Commission de protection du territoire Agricole du Québec</i>
CQLR	Compilation of Québec Laws and Regulations
CRM	Certified Reference Materials
CTEU-9	Leaching test under near-neutral conditions (Québec method)
Cu	Copper
CVAA	Acute Aquatic Life Criteria
CVAC	Chronic Aquatic Life Criteria
d	day
DDH	Diamond Drill Hole
dm	Decameter
ECCC	Environment and Climate Change Canada
EER	Exclusive Exploration Right
EIA	Environmental Impact Assessment
EIARP	Environmental Impact Assessment and Review Procedure
EIJBRG	Environmental Impact Assessment and Review Procedure
EIS	Environmental Impact Study
EM	Electromagnetic
Ep	Epidote
EQA	Environment Quality Act
ESA	Environmental Site Assessment



Abbreviation or Symbol	Description
Evomine	Evomine Consulting Inc.
FS	Feasibility Study
ft	Foot
FW	Footwall
g	Gram
G	Billion
g	Gram
g/L	Gram per litre
Ga	Billion years
Ga	Billion years
GCMP	Ground Control Management Program
Geodoz	Geodoz Conseil Inc.
GESTIM	Gestion des Titres Miniers
GMR	Gross Metal Revenue
GPS	Global Positioning System
gpt, g/t	Grams per metric tonne
h	Hour
ha	Hectare
ha	Hectare
hp	Horse Power
hpd, h/d	Hours per day
HW	Hanging wall
IAA	Impact Assessment Act
ICP	Inductively Coupled Plasma
ID3	Inverse Distance cubed
in	Inch
IP	Induce Polarization
IREM	Institut de Recherche en Exploration Minérale
IRR	Internal Rate of Return
ISO	International Organization for Standardization
JBNQA	James Bay and Northern Québec Agreement
k	Thousand
K80	Dimension that represents the 80 th percentil of a size distribution
kg	Kilogram
kg/t	Kilogram per tonne
km	Kilometer
km/h	Kilometer per hour
km/h	Kilometer per hour
km ²	Square kilometer
koz	Thousand Ounces
koz	Thousand Ounces

Abbreviation or Symbol	Description
kV	Kilovolt
kVA	Kilovolt-ampere
kW	Kilowatt
kWh	Kilowatt-hour
L	Liter
LHD	Load-Haul-Dump
LOI	Letter of Intent
LOM	Life of Mine
m	Meter
M.T. S	Mineral Title Status
m ²	Square meter
m ²	Square meter
m ³	Cubic Meter
m ³ /h	Cubic meters per hour
m ³ /year	Cubic meters per year
Ma	Million years
MA100 Lix Com 1.1	Leaching analytical method
MA110 ACISOL 1.0	ABA analytical method recognized by MELCCFP
MA200	Analytical method for extractable metals
MAG	Magnetometric, Magnetometer
MDDELCC	<i>Ministère du Développement Durable, de l'Environnement et de la Lutte contre les changements climatiques</i>
MDMER	Metal and Diamond Mining Effluent Regulations
MELCC	<i>Ministère de l'Environnement et de la Lutte contre les changements climatiques</i>
MELCCFP	Ministry of the Environment, the Fight Against Climate Change, Wildlife and Parks
Mia	Grindability for size reduction down to 750 µm
Mib	Grindability for size reduction from 750 µm
Mic	Grindability for crusher circuit
ml	Milliliter
ML-ARD	Metal Leaching and Acid Rock Drainage
mm	Millimeter
Moz	Million Ounces
MRCVO	<i>Municipalité régionale de comté de la Vallée-de-l'Or</i>
MRE	Mineral Resource Estimate
MRNF	Ministère des Ressources Naturelles et des Forêts
MS	Massive Sulphide
Mtpa	Million tonnes per annum
Mtpa	Million tonnes per annum
NI 43-101, 43-101	National Instrument 43-101
NN	Nearest Neighbor
NP	Neutralization Potential



Abbreviation or Symbol	Description
NPAG	Non-Potentially Acid Generating
NPP	Net Neutralization Potential
NPR	Neutralization Potential Ratio
NPV	Net Present Value
NQ	Drill Core Diameter
NS	North-South
NSR	Net Smelter Return
OP, O/P	Open pit
OP, O/P	Open pit
OPEX	Operating Expenses
OREAS	Ore Research & Exploration Pty Ltd. Assay standards
oz	Troy Ounce
PAG	Potentially Acid Generating
Pb	Lead
PEA	Preliminary Economic Assessment
PFS	Pre-Feasibility Study
pH	Potential of Hydrogen
PLR	PLR Resources Inc.
PM _{2.5}	Particulate matter (fine)
Po	Pyrrhotite
Po	Pyrrhotite
ppb	Parts per Billion
ppb	Parts per Billion
ppm	Parts per Million
ppm	Parts per Million
Py	Pyrite
QA/QC	Quality Assurance/Quality Control
QA/QC	Quality Assurance/Quality Control
QEM-ARMS	Quantitative Evaluation of Materials by Automated Rapid Mineral Scan
QEMSCAN	Quantitative Evaluation of Materials by Scanning Electron Microscopy
QP	Qualified Person
RA	Access Ramp
RC	Reverse Circulation
RCM	Regional County Municipality
RLRQ	<i>Recueil des lois et des règlements du Québec</i>
RQD	Rock Quality Designation
RSQAQ	<i>Réseau de surveillance de la qualité de l'air du Québec</i>
SCSE	Sag Circuit Specific Energy
SD	Standard deviation
SIGEOM	Système d'Information Géominière du Québec
SMC	Sag Mill Comminution test

Abbreviation or Symbol	Description
Soutex	Soutex Inc.
Sp	Sphalerite
SPLP	Synthetic Precipitation Leaching Procedure
SQUID	Superconducting Quantum Interference Device
Ssulf	Sulfide sulfur
Stotal	Total sulfur
SWQC	Surface Water Quality Criteria
t	Tonne (metric tonne)
TCLP	Toxicity Characteristic Leaching Procedure
TDEM	Time-Domain Electromagnetic
TMF	Tailings Management Facility
Tpd, t/d	Tonnes per day
tph, t/h	Tonnes per hour
UG, U/G	Underground
US\$ or USD	United States dollar
UTM	Universal Transverse Mercator
V	Volt
VG	Visible Gold
VMS	Volcanogenic Massive Sulfide
WTP	Water Treatment Plant
Y	Year

2.4 Acknowledgements

The Report Authors would like to acknowledge the support they received from the issuer's employees and other collaborators during this assignment. Their collaboration is greatly appreciated. The Report benefitted from the input of the following individuals:

- Philippe Cloutier, President and CEO - Cartier
- Ronan Derooff, Vice President Exploration – Cartier
- Myrzah Tavares Bello, Director - Cartier



3 RELIANCE ON OTHER EXPERTS

This Technical Report has been prepared by Evomine Consulting, Geodoz Conseil, Soutex and PLR Resources. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to the QPs at the time of the preparation of this Report.
- Assumptions, conditions, and qualifications as set forth in this Report.
- Data, reports, and opinions supplied by Cartier Resources and other third-party sources.

The QPs of this Technical Report believe that the basic assumptions contained in the information indicated above are factual and accurate and that the interpretations are reasonable. The QPs of this Technical Report have, to the extent applicable, relied on this data and have no reason to believe that any material facts have been withheld. The QPs of this Technical Report have taken all appropriate steps, in their professional judgement, to ensure that the work, information, or advice from the above indicated information is sound and the QPs do not disclaim any responsibility for this Technical Report.

The QPs have relied upon reports, information sources, and opinions provided by the issuer and outside experts regarding the Project's mineral rights, surface rights, agreements, royalties, and commodity markets. The issuer has indicated that there are no known litigations potentially affecting the Project.

The issuer has reviewed a draft copy of the Report for factual errors. Any changes made as a result of these reviews did not involve any alteration to the conclusions made. Hence, the statements and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are neither false nor misleading at the date of this Report.

3.1 Mineral Tenure and Surface Rights

The issuer supplied information on mining titles. The QPs consulted the Government of Quebec's online claim management system at <https://gestim.mines.gouv.qc.ca> for the latest ownership and mining title status. Although the QPs have reviewed the option agreements and claim status, QPs are not qualified to express any legal opinion concerning the titles, current ownership or possible litigations. A description of such agreements and the Project and ownership thereof is provided for general information only. In this regard, the QPs have relied on information supplied by the issuer and the work of experts they understand to be appropriately qualified. This information supports





Chapter 4 (Property Description and Location).



4 PROPERTY DESCRIPTION AND LOCATION

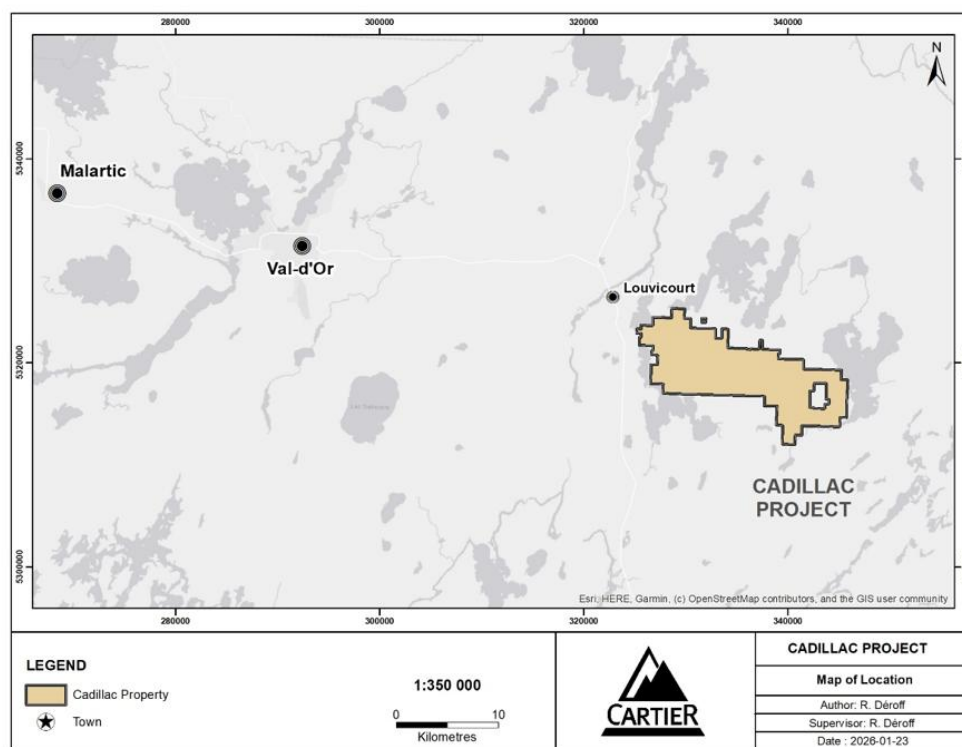
The Cadillac Project is located approximately 35 km east-southeast of the town of Val-d'Or, in the Province of Québec, Canada and approximately 5 km southeast of the municipality of Louvicourt (Figure 4-1). The Project is accessible year-round via paved roads and secondary gravel roads.

The coordinates for the approximate centre of the Project are latitude 48°00' N and longitude 77°06' W (343350E and 5318430N: NAD 83 / UTM Zone 18N) on NTS map sheets 32C02, and 32N14.

The outlines of the claims were obtained from GESTIM Plus, the online mining title management system of the ministère des Ressources naturelles et des Forêts ("MRNF"):

https://gestim.mines.gouv.qc.ca/MRN_GestimP_Presentation/ODM02101_login.aspx

Figure 4-1 : Project location



4.1 Mineral Tenure

Cartier controls a group of 281 mining titles covering a total of 11,596 ha. This information is current as of the effective date of this Report. Figure 4-2 shows the mining titles of the Cadillac Project.

The mining titles are recorded under Cartier Resources Inc. and are in good standing as of the

effective date of this Report (Table 4-1).

Figure 4-2 : Mining Titles

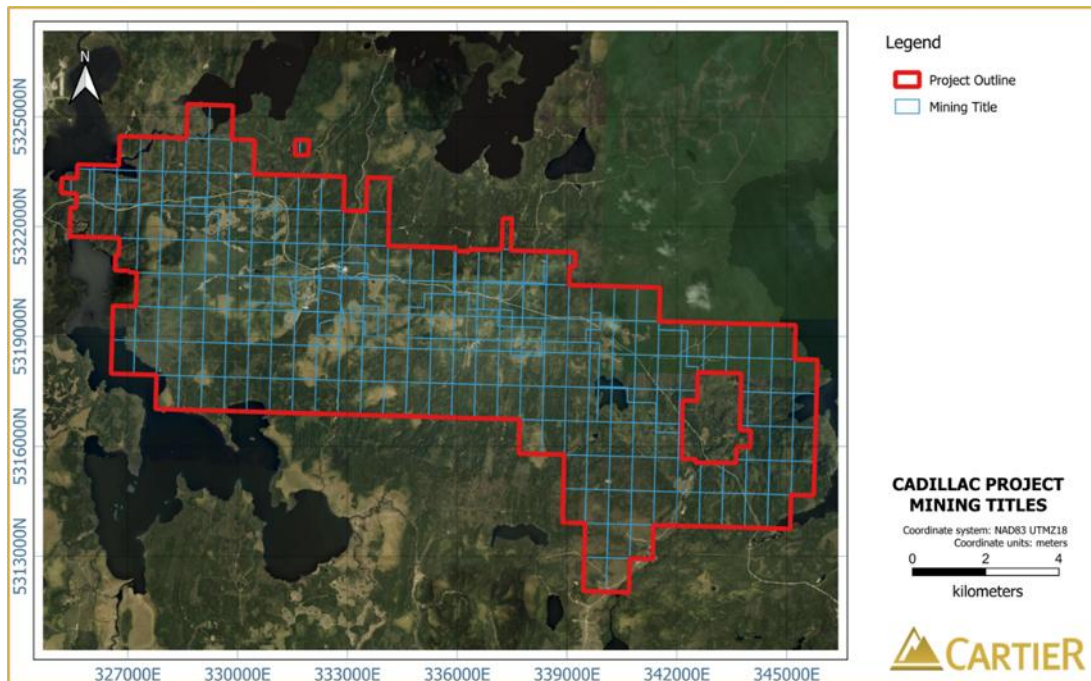


Table 4-1 : Details of Mining Titles

Title No	Title Type	Title Status	SNRC	Area (ha)	Registration Date	Expiration Date	Credit	Owner	Royalties
2456677	CDC	Active	32C03	57.59	2016-08-09	2028-08-08	\$ 51	Ressources Cartier inc.	2% NSR to Dean Boudrias
2456678	CDC	Active	32C03	57.57	2016-08-09	2028-08-08	\$ -	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2456679	CDC	Active	32C03	57.57	2016-08-09	2028-08-08	\$ 197,099	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2456680	CDC	Active	32C03	57.57	2016-08-09	2028-08-08	\$ 137	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2438798	CDC	Active	32C03	57.60	2016-05-04	2028-08-14	\$ 76,254	Ressources Cartier inc.	3%GMR Globex Mining Enterprises Inc.
2438799	CDC	Active	32C03	57.60	2016-05-04	2028-08-14	\$ 191,578	Ressources Cartier inc.	3%GMR Globex Mining Enterprises Inc.
2438800	CDC	Active	32C03	57.60	2016-05-04	2028-08-14	\$ 125,600	Ressources Cartier inc.	3%GMR Globex Mining Enterprises Inc.
2438801	CDC	Active	32C03	50.80	2016-05-04	2028-08-14	\$ 68,505	Ressources Cartier inc.	3%GMR Globex Mining Enterprises Inc.
2438802	CDC	Active	32C03	18.51	2016-05-04	2028-08-14	\$ 253,904	Ressources Cartier inc.	3%GMR Globex Mining Enterprises Inc.
2438803	CDC	Active	32C03	2.23	2016-05-04	2028-08-14	\$ -	Ressources Cartier inc.	3%GMR Globex Mining Enterprises Inc.
2438804	CDC	Active	32C03	39.36	2016-05-04	2028-08-14	\$ 53,708	Ressources Cartier inc.	3%GMR Globex Mining Enterprises Inc.
2438805	CDC	Active	32C03	7.15	2016-05-04	2028-08-14	\$ 6,927	Ressources Cartier inc.	3%GMR Globex Mining Enterprises Inc.
2438806	CDC	Active	32C03	5.03	2016-05-04	2028-08-14	\$ 2,882	Ressources Cartier inc.	3%GMR Globex Mining Enterprises Inc.
2438807	CDC	Active	32C03	34.64	2016-05-04	2028-08-14	\$ 45,523	Ressources Cartier inc.	3%GMR Globex Mining Enterprises Inc.
2438808	CDC	Active	32C03	5.61	2016-05-04	2028-08-14	\$ 4,348	Ressources Cartier inc.	3%GMR Globex Mining Enterprises Inc.
2438809	CDC	Active	32C03	6.93	2016-05-04	2028-08-14	\$ 7,282	Ressources Cartier inc.	3%GMR Globex Mining Enterprises Inc.

Title No	Title Type	Title Status	SNRC	Area (ha)	Registration Date	Expiration Date	Credit	Owner	Royalties
2438810	CDC	Active	32C03	1.76	2016-05-04	2028-08-14	\$ -	Ressources Cartier inc.	3%GMR Globex Mining Enterprises Inc.
2438811	CDC	Active	32C03	33.42	2016-05-04	2028-08-14	\$ 42,152	Ressources Cartier inc.	3%GMR Globex Mining Enterprises Inc.
2457890	CDC	Active	32C03	57.57	2016-08-17	2028-08-16	\$ -	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2457891	CDC	Active	32C03	57.57	2016-08-17	2028-08-16	\$ -	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2458268	CDC	Active	32C03	57.58	2016-08-17	2028-08-16	\$ 339	Ressources Cartier inc.	2% NSR to Dean Boudrias
2458269	CDC	Active	32C03	57.58	2016-08-17	2028-08-16	\$ -	Ressources Cartier inc.	2% NSR to Dean Boudrias
2458270	CDC	Active	32C03	57.58	2016-08-17	2028-08-16	\$ -	Ressources Cartier inc.	2% NSR to Dean Boudrias
2458271	CDC	Active	32C03	57.57	2016-08-17	2028-08-16	\$ -	Ressources Cartier inc.	2% NSR to Dean Boudrias
2458272	CDC	Active	32C03	57.57	2016-08-17	2028-08-16	\$ -	Ressources Cartier inc.	2% NSR to Dean Boudrias
2461488	CDC	Active	31N14	57.62	2016-09-08	2028-09-07	\$ 274,156	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2461489	CDC	Active	31N14	57.62	2016-09-08	2028-09-07	\$ 854	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2461490	CDC	Active	31N14	57.62	2016-09-08	2028-09-07	\$ 1,296	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2461491	CDC	Active	32C03	57.61	2016-09-08	2028-09-07	\$ 2,370	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2461492	CDC	Active	32C03	57.61	2016-09-08	2028-09-07	\$ 125,164	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2461493	CDC	Active	32C03	57.61	2016-09-08	2028-09-07	\$ 3,873	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2461494	CDC	Active	32C03	57.61	2016-09-08	2028-09-07	\$ 3,596	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2461495	CDC	Active	32C03	57.61	2016-09-08	2028-09-07	\$ 51,571	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2466091	CDC	Active	31N14	57.62	2016-10-17	2028-10-16	\$ -	Ressources Cartier inc.	2% NSR to Vorenius Metal Corp
2466092	CDC	Active	32C03	57.61	2016-10-17	2028-10-16	\$ -	Ressources Cartier inc.	2% NSR to Vorenius Metal Corp
2438058	CDC	Active	32C03	57.58	2016-04-08	2028-10-31	\$ 3,064	Ressources Cartier inc.	2% NSR to Harfang Exploration Inc.
2438059	CDC	Active	32C03	57.58	2016-04-08	2028-10-31	\$ 3,650	Ressources Cartier inc.	2% NSR to Harfang Exploration Inc.
2438060	CDC	Active	32C03	57.58	2016-04-08	2028-10-31	\$ 3,408	Ressources Cartier inc.	2% NSR to Harfang Exploration Inc.
2438061	CDC	Active	32C03	57.58	2016-04-08	2028-10-31	\$ 4,213	Ressources Cartier inc.	2% NSR to Harfang Exploration Inc.
2438062	CDC	Active	32C03	57.58	2016-04-08	2028-10-31	\$ 5,272	Ressources Cartier inc.	2% NSR to Harfang Exploration Inc.
2438063	CDC	Active	32C03	47.10	2016-04-08	2028-10-31	\$ -	Ressources Cartier inc.	2% NSR to Harfang Exploration Inc.
2438064	CDC	Active	32C03	1.02	2016-04-08	2028-10-31	\$ -	Ressources Cartier inc.	2% NSR to Harfang Exploration Inc.
2438065	CDC	Active	32C03	34.49	2016-04-08	2028-10-31	\$ -	Ressources Cartier inc.	2% NSR to Harfang Exploration Inc.
2438066	CDC	Active	32C03	57.58	2016-04-08	2028-10-31	\$ 1,981	Ressources Cartier inc.	2% NSR to Harfang Exploration Inc.
2438067	CDC	Active	32C03	57.58	2016-04-08	2028-10-31	\$ 2,626	Ressources Cartier inc.	2% NSR to Harfang Exploration Inc.
2468029	CDC	Active	31N14	57.64	2016-11-07	2028-11-06	\$ 30,030	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2468030	CDC	Active	31N14	57.64	2016-11-07	2028-11-06	\$ 198,851	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2468031	CDC	Active	31N14	57.64	2016-11-07	2028-11-06	\$ -	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2468032	CDC	Active	31N14	57.64	2016-11-07	2028-11-06	\$ -	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2468033	CDC	Active	31N14	57.64	2016-11-07	2028-11-06	\$ 96,799	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2468034	CDC	Active	31N14	57.64	2016-11-07	2028-11-06	\$ 8,512	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2468035	CDC	Active	31N14	57.64	2016-11-07	2028-11-06	\$ 10,123	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2468036	CDC	Active	31N14	57.63	2016-11-07	2028-11-06	\$ -	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2468037	CDC	Active	31N14	57.63	2016-11-07	2028-11-06	\$ 84,631	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2468038	CDC	Active	31N14	57.63	2016-11-07	2028-11-06	\$ 1,724	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.



Title No	Title Type	Title Status	SNRC	Area (ha)	Registration Date	Expiration Date	Credit	Owner	Royalties
2468039	CDC	Active	31N14	57.63	2016-11-07	2028-11-06	\$ 2,125	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2468040	CDC	Active	31N14	57.63	2016-11-07	2028-11-06	\$ 184,395	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2468041	CDC	Active	31N14	57.62	2016-11-07	2028-11-06	\$ -	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2468042	CDC	Active	31N14	57.62	2016-11-07	2028-11-06	\$ 364	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2468043	CDC	Active	31N14	57.62	2016-11-07	2028-11-06	\$ 3,191	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2470586	CDC	Active	31N14	57.62	2016-12-07	2028-12-06	\$ 924	Ressources Cartier inc.	2% NSR to Vorenius Metal Corp
2438935	CDC	Active	32C03	3.24	2016-04-27	2028-12-12	\$ 52,203	Ressources Cartier inc.	3%GMR Globex Mining Enterprises Inc.
2438936	CDC	Active	32C03	24.05	2016-04-27	2028-12-12	\$ 111,747	Ressources Cartier inc.	3%GMR Globex Mining Enterprises Inc.
2438937	CDC	Active	32C03	3.85	2016-04-27	2028-12-12	\$ -	Ressources Cartier inc.	3%GMR Globex Mining Enterprises Inc.
2471188	CDC	Active	32C03	1.40	2016-12-22	2028-12-21	\$ -	Ressources Cartier inc.	2% NSR Gilbert Lamothe & Victor Cantore
2471189	CDC	Active	32C03	30.83	2016-12-22	2028-12-21	\$ 27,114	Ressources Cartier inc.	2% NSR Gilbert Lamothe & Victor Cantore
2471190	CDC	Active	32C03	11.85	2016-12-22	2028-12-21	\$ -	Ressources Cartier inc.	2% NSR Gilbert Lamothe & Victor Cantore
2471191	CDC	Active	32C03	3.86	2016-12-22	2028-12-21	\$ -	Ressources Cartier inc.	2% NSR Gilbert Lamothe & Victor Cantore
2471192	CDC	Active	32C03	53.73	2016-12-22	2028-12-21	\$ -	Ressources Cartier inc.	2% NSR Gilbert Lamothe & Victor Cantore
2471193	CDC	Active	32C03	54.35	2016-12-22	2028-12-21	\$ -	Ressources Cartier inc.	2% NSR Gilbert Lamothe & Victor Cantore
2471194	CDC	Active	32C03	54.27	2016-12-22	2028-12-21	\$ -	Ressources Cartier inc.	2% NSR Gilbert Lamothe & Victor Cantore
2471195	CDC	Active	32C03	50.14	2016-12-22	2028-12-21	\$ -	Ressources Cartier inc.	2% NSR Gilbert Lamothe & Victor Cantore
2471196	CDC	Active	32C03	55.70	2016-12-22	2028-12-21	\$ -	Ressources Cartier inc.	2% NSR Gilbert Lamothe & Victor Cantore
2471197	CDC	Active	32C03	5.74	2016-12-22	2028-12-21	\$ -	Ressources Cartier inc.	2% NSR Gilbert Lamothe & Victor Cantore
2471198	CDC	Active	32C03	55.47	2016-12-22	2028-12-21	\$ -	Ressources Cartier inc.	2% NSR Gilbert Lamothe & Victor Cantore
2471200	CDC	Active	32C03	21.30	2016-12-22	2028-12-21	\$ -	Ressources Cartier inc.	2% NSR Gilbert Lamothe & Victor Cantore
2471201	CDC	Active	32C03	10.97	2016-12-22	2028-12-21	\$ -	Ressources Cartier inc.	2% NSR Gilbert Lamothe & Victor Cantore
2471202	CDC	Active	32C03	41.19	2016-12-22	2028-12-21	\$ 412	Ressources Cartier inc.	2% NSR Gilbert Lamothe & Victor Cantore
2472374	CDC	Active	32C03	11.30	2017-01-09	2029-01-08	\$ -	Ressources Cartier inc.	1% NSR Marc De Keyser & Raymond Chartrand
2472375	CDC	Active	32C03	10.47	2017-01-09	2029-01-08	\$ -	Ressources Cartier inc.	1% NSR Marc De Keyser & Raymond Chartrand
2480254	CDC	Active	31N14	57.61	2017-02-23	2027-02-22	\$ 440	Ressources Cartier inc.	2% NSR to Canadian Mining House
2481223	CDC	Active	31N14	57.65	2017-02-27	2027-02-26	\$ 18,633	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2481224	CDC	Active	31N14	57.65	2017-02-27	2027-02-26	\$ 10,916	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2481228	CDC	Active	31N14	57.65	2017-02-27	2027-02-26	\$ 351,679	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2481243	CDC	Active	31N14	57.63	2017-02-27	2027-02-26	\$ 666	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2481244	CDC	Active	31N14	57.63	2017-02-27	2027-02-26	\$ 188	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2481245	CDC	Active	31N14	57.63	2017-02-27	2027-02-26	\$ 85	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2481246	CDC	Active	31N14	57.63	2017-02-27	2027-02-26	\$ 90,163	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2481247	CDC	Active	31N14	57.63	2017-02-27	2027-02-26	\$ 736	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2481248	CDC	Active	31N14	57.63	2017-02-27	2027-02-26	\$ 954	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2481249	CDC	Active	31N14	57.63	2017-02-27	2027-02-26	\$ 182	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2481250	CDC	Active	31N14	57.63	2017-02-27	2027-02-26	\$ 519	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2481251	CDC	Active	31N14	57.63	2017-02-27	2027-02-26	\$ 422	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2481252	CDC	Active	31N14	57.63	2017-02-27	2027-02-26	\$ -	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.



Title No	Title Type	Title Status	SNRC	Area (ha)	Registration Date	Expiration Date	Credit	Owner	Royalties
2481253	CDC	Active	31N14	57.63	2017-02-27	2027-02-26	\$ -	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2481254	CDC	Active	31N14	57.63	2017-02-27	2027-02-26	\$ -	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2481271	CDC	Active	31N14	57.66	2017-02-27	2027-02-26	\$ 12,188	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2481272	CDC	Active	31N14	57.66	2017-02-27	2027-02-26	\$ 10,012	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2481273	CDC	Active	31N14	57.66	2017-02-27	2027-02-26	\$ 7,157	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2481283	CDC	Active	31N14	57.65	2017-02-27	2027-02-26	\$ 138,266	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2481297	CDC	Active	31N14	57.67	2017-02-27	2027-02-26	\$ 9,429	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2481298	CDC	Active	31N14	57.67	2017-02-27	2027-02-26	\$ 9,947	Ressources Cartier inc.	1% NSR to Chalice Gold Mines Ont Inc.
2439460	CDC	Active	32C03	57.61	2016-04-22	2027-02-28	\$ 9,785	Ressources Cartier inc.	1% NSR to Trip Flag Precious Metals
2439464	CDC	Active	32C03	1.88	2016-04-22	2027-02-28	\$ -	Ressources Cartier inc.	1% NSR to Trip Flag Precious Metals
2439467	CDC	Active	32C03	57.61	2016-04-22	2027-02-28	\$ 11,177	Ressources Cartier inc.	1% NSR to Trip Flag Precious Metals
2439469	CDC	Active	32C03	19.05	2016-04-22	2027-02-28	\$ 1,807,137	Ressources Cartier inc.	1% NSR to Trip Flag Precious Metals
2439474	CDC	Active	32C03	0.93	2016-04-22	2027-02-28	\$ -	Ressources Cartier inc.	1% NSR to Trip Flag Precious Metals
2439476	CDC	Active	32C03	12.27	2016-04-22	2027-02-28	\$ 1,281	Ressources Cartier inc.	1% NSR to Trip Flag Precious Metals
2439479	CDC	Active	32C03	57.61	2016-04-22	2027-02-28	\$ 523,593	Ressources Cartier inc.	1% NSR to Trip Flag Precious Metals
2439480	CDC	Active	32C03	35.42	2016-04-22	2027-02-28	\$ 27,718	Ressources Cartier inc.	1% NSR to Trip Flag Precious Metals
2439487	CDC	Active	32C03	35.24	2016-04-22	2027-02-28	\$ 4,903,635	Ressources Cartier inc.	1% NSR to Trip Flag Precious Metals
2438130	CDC	Active	32C03	0.54	2016-04-19	2027-05-01	\$ -	Ressources Cartier inc.	2% NSR to Harfang Exploration Inc.
2438131	CDC	Active	32C03	1.07	2016-04-19	2027-05-01	\$ -	Ressources Cartier inc.	2% NSR to Harfang Exploration Inc.
2438132	CDC	Active	32C03	4.88	2016-04-19	2027-05-01	\$ -	Ressources Cartier inc.	2% NSR to Harfang Exploration Inc.
2438133	CDC	Active	32C03	8.90	2016-04-19	2027-05-01	\$ -	Ressources Cartier inc.	2% NSR to Harfang Exploration Inc.
2438103	CDC	Active	32C03	57.58	2016-04-08	2027-05-02	\$ -	Ressources Cartier inc.	2% NSR to Harfang Exploration Inc.
2438104	CDC	Active	32C03	57.57	2016-04-08	2027-05-02	\$ -	Ressources Cartier inc.	2% NSR to Harfang Exploration Inc.
2491522	CDC	Active	32C03	57.58	2017-05-04	2027-05-03	\$ 4,282	Ressources Cartier inc.	1% NSR to Marc De Keyser
2385084	CDC	Active	31N14	23.67	2013-05-13	2027-05-12	\$ 121,068	Ressources Cartier inc.	
2437791	CDC	Active	32C03	57.60	2016-04-06	2027-06-27	\$ 5,311	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437792	CDC	Active	32C03	57.60	2016-04-06	2027-06-27	\$ 2,378	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437793	CDC	Active	32C03	57.61	2016-04-06	2027-06-27	\$ 8,342	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437794	CDC	Active	31N14	57.62	2016-04-06	2027-06-27	\$ 7,709	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437795	CDC	Active	31N14	57.62	2016-04-06	2027-06-27	\$ 7,707	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437796	CDC	Active	31N14	57.63	2016-04-06	2027-06-27	\$ 6,957	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437797	CDC	Active	31N14	23.58	2016-04-06	2027-06-27	\$ 2,876	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437798	CDC	Active	31N14	6.94	2016-04-06	2027-06-27	\$ -	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437799	CDC	Active	32C03	43.03	2016-04-06	2027-06-27	\$ 4,460	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437800	CDC	Active	32C03	57.35	2016-04-06	2027-06-27	\$ 7,907	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437801	CDC	Active	32C03	52.57	2016-04-06	2027-06-27	\$ -	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437802	CDC	Active	31N14	32.74	2016-04-06	2027-06-27	\$ 969	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437803	CDC	Active	31N14	0.30	2016-04-06	2027-06-27	\$ -	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437804	CDC	Active	31N14	18.32	2016-04-06	2027-06-27	\$ 149	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.



Title No	Title Type	Title Status	SNRC	Area (ha)	Registration Date	Expiration Date	Credit	Owner	Royalties
2437805	CDC	Active	31N14	56.27	2016-04-06	2027-06-27	\$ 7,012	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437806	CDC	Active	32C03	4.43	2016-04-06	2027-06-27	\$ -	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437807	CDC	Active	32C03	43.39	2016-04-06	2027-06-27	\$ 4,252	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437808	CDC	Active	31N14	14.98	2016-04-06	2027-06-27	\$ 1,891	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437809	CDC	Active	31N14	35.21	2016-04-06	2027-06-27	\$ -	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437810	CDC	Active	32C03	11.57	2016-04-06	2027-06-27	\$ 403	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437811	CDC	Active	31N14	1.98	2016-04-06	2027-06-27	\$ -	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2431503	CDC	Active	32C03	55.78	2015-07-23	2027-07-22	\$ 7,373,007	Ressources Cartier inc.	1% NSR to Trip Flag Precious Metals
2438140	CDC	Active	32C03	57.60	2016-04-22	2027-08-01	\$ 143,133	Ressources Cartier inc.	
2438141	CDC	Active	32C03	57.59	2016-04-22	2027-08-01	\$ 351,022	Ressources Cartier inc.	
2438142	CDC	Active	32C03	57.59	2016-04-22	2027-08-01	\$ 258,510	Ressources Cartier inc.	
2438143	CDC	Active	32C03	57.59	2016-04-22	2027-08-01	\$ 1,621	Ressources Cartier inc.	
2438144	CDC	Active	32C03	57.59	2016-04-22	2027-08-01	\$ 334	Ressources Cartier inc.	
2438145	CDC	Active	32C03	57.60	2016-04-22	2027-08-01	\$ 191,146	Ressources Cartier inc.	
2438146	CDC	Active	32C03	57.59	2016-04-22	2027-08-01	\$ 108,811	Ressources Cartier inc.	
2438147	CDC	Active	32C03	57.59	2016-04-22	2027-08-01	\$ 118,389	Ressources Cartier inc.	
2438148	CDC	Active	32C03	57.59	2016-04-22	2027-08-01	\$ 7,194	Ressources Cartier inc.	
2438149	CDC	Active	31N14	57.62	2016-04-22	2027-08-01	\$ 295,597	Ressources Cartier inc.	
2438150	CDC	Active	31N14	57.62	2016-04-22	2027-08-01	\$ 6,723	Ressources Cartier inc.	
2438151	CDC	Active	31N14	57.62	2016-04-22	2027-08-01	\$ 30,771	Ressources Cartier inc.	
2438152	CDC	Active	31N14	57.62	2016-04-22	2027-08-01	\$ 7,720	Ressources Cartier inc.	
2438153	CDC	Active	31N14	57.62	2016-04-22	2027-08-01	\$ 7,612	Ressources Cartier inc.	
2438154	CDC	Active	32C03	23.10	2016-04-22	2027-08-01	\$ 3,281	Ressources Cartier inc.	
2438155	CDC	Active	31N14	57.62	2016-04-22	2027-08-01	\$ 12,315	Ressources Cartier inc.	
2438156	CDC	Active	32C03	57.59	2016-04-22	2027-08-01	\$ 118,956	Ressources Cartier inc.	
2438157	CDC	Active	32C03	23.24	2016-04-22	2027-08-01	\$ 115,996	Ressources Cartier inc.	
2438158	CDC	Active	32C03	18.25	2016-04-22	2027-08-01	\$ 2,263	Ressources Cartier inc.	
2438159	CDC	Active	32C03	21.14	2016-04-22	2027-08-01	\$ 59,908	Ressources Cartier inc.	
2438160	CDC	Active	32C03	6.80	2016-04-22	2027-08-01	\$ 45	Ressources Cartier inc.	
2438161	CDC	Active	32C03	8.49	2016-04-22	2027-08-01	\$ -	Ressources Cartier inc.	
2438162	CDC	Active	32C03	57.59	2016-04-22	2027-08-01	\$ 6,655	Ressources Cartier inc.	
2438163	CDC	Active	32C03	30.41	2016-04-22	2027-08-01	\$ -	Ressources Cartier inc.	
2438164	CDC	Active	31N14	57.62	2016-04-22	2027-08-01	\$ 2,383	Ressources Cartier inc.	
2438165	CDC	Active	32C03	13.26	2016-04-22	2027-08-01	\$ 57,056	Ressources Cartier inc.	
2438166	CDC	Active	32C03	20.04	2016-04-22	2027-08-01	\$ -	Ressources Cartier inc.	
2438167	CDC	Active	31N14	57.62	2016-04-22	2027-08-01	\$ 6,127	Ressources Cartier inc.	
2438168	CDC	Active	32C03	48.47	2016-04-22	2027-08-01	\$ 328,093	Ressources Cartier inc.	
2438169	CDC	Active	32C03	54.35	2016-04-22	2027-08-01	\$ 17,183	Ressources Cartier inc.	
2438170	CDC	Active	32C03	18.07	2016-04-22	2027-08-01	\$ 2,009	Ressources Cartier inc.	



Title No	Title Type	Title Status	SNRC	Area (ha)	Registration Date	Expiration Date	Credit	Owner	Royalties
2438171	CDC	Active	32C03	57.60	2016-04-22	2027-08-01	\$ 7,961	Ressources Cartier inc.	
2438172	CDC	Active	32C03	22.36	2016-04-22	2027-08-01	\$ 1,567,655	Ressources Cartier inc.	
2438173	CDC	Active	32C03	7.64	2016-04-22	2027-08-01	\$ -	Ressources Cartier inc.	
2438174	CDC	Active	32C03	57.60	2016-04-22	2027-08-01	\$ 7,993	Ressources Cartier inc.	
2438175	CDC	Active	31N14	23.74	2016-04-22	2027-08-01	\$ 3,640	Ressources Cartier inc.	
2438176	CDC	Active	32C03	57.59	2016-04-22	2027-08-01	\$ 7,801	Ressources Cartier inc.	
2438177	CDC	Active	32C03	1.05	2016-04-22	2027-08-01	\$ -	Ressources Cartier inc.	
2438178	CDC	Active	32C03	57.61	2016-04-22	2027-08-01	\$ 76,165	Ressources Cartier inc.	
2438179	CDC	Active	32C03	18.12	2016-04-22	2027-08-01	\$ 123,936	Ressources Cartier inc.	
2438180	CDC	Active	32C03	29.17	2016-04-22	2027-08-01	\$ 362	Ressources Cartier inc.	
2438181	CDC	Active	32C03	5.45	2016-04-22	2027-08-01	\$ -	Ressources Cartier inc.	
2438182	CDC	Active	32C03	57.60	2016-04-22	2027-08-01	\$ 271,211	Ressources Cartier inc.	
2438183	CDC	Active	32C03	49.40	2016-04-22	2027-08-01	\$ 177,880	Ressources Cartier inc.	
2438184	CDC	Active	31N14	57.62	2016-04-22	2027-08-01	\$ 4,931	Ressources Cartier inc.	
2438185	CDC	Active	32C03	18.18	2016-04-22	2027-08-01	\$ 33,239	Ressources Cartier inc.	
2438186	CDC	Active	32C03	45.34	2016-04-22	2027-08-01	\$ 1,120,163	Ressources Cartier inc.	
2438187	CDC	Active	32C03	49.51	2016-04-22	2027-08-01	\$ 2,344,245	Ressources Cartier inc.	
2438188	CDC	Active	32C03	45.26	2016-04-22	2027-08-01	\$ 5,503	Ressources Cartier inc.	
2438189	CDC	Active	32C03	57.60	2016-04-22	2027-08-01	\$ 81,214	Ressources Cartier inc.	
2438190	CDC	Active	32C03	49.49	2016-04-22	2027-08-01	\$ 2,635,068	Ressources Cartier inc.	
2438191	CDC	Active	32C03	57.58	2016-04-22	2027-08-01	\$ -	Ressources Cartier inc.	
2438192	CDC	Active	32C03	18.61	2016-04-22	2027-08-01	\$ 4,183	Ressources Cartier inc.	
2438193	CDC	Active	32C03	57.60	2016-04-22	2027-08-01	\$ 277,222	Ressources Cartier inc.	
2438194	CDC	Active	32C03	56.91	2016-04-22	2027-08-01	\$ 174,017	Ressources Cartier inc.	
2438195	CDC	Active	31N14	51.88	2016-04-22	2027-08-01	\$ 6,862	Ressources Cartier inc.	
2438196	CDC	Active	32C03	18.53	2016-04-22	2027-08-01	\$ 2,322	Ressources Cartier inc.	
2438197	CDC	Active	32C03	57.58	2016-04-22	2027-08-01	\$ -	Ressources Cartier inc.	
2438198	CDC	Active	32C03	3.13	2016-04-22	2027-08-01	\$ -	Ressources Cartier inc.	
2438199	CDC	Active	32C03	37.53	2016-04-22	2027-08-01	\$ 11,893	Ressources Cartier inc.	
2438200	CDC	Active	32C03	38.56	2016-04-22	2027-08-01	\$ 1,562,446	Ressources Cartier inc.	
2438201	CDC	Active	32C03	1.82	2016-04-22	2027-08-01	\$ 77,375	Ressources Cartier inc.	
2438202	CDC	Active	32C03	53.39	2016-04-22	2027-08-01	\$ 112,405	Ressources Cartier inc.	
2438203	CDC	Active	32C03	53.73	2016-04-22	2027-08-01	\$ -	Ressources Cartier inc.	
2438204	CDC	Active	32C03	19.94	2016-04-22	2027-08-01	\$ 671,118	Ressources Cartier inc.	
2438205	CDC	Active	32C03	1.35	2016-04-22	2027-08-01	\$ -	Ressources Cartier inc.	
2438206	CDC	Active	32C03	57.60	2016-04-22	2027-08-01	\$ 371,888	Ressources Cartier inc.	
2438207	CDC	Active	32C03	54.18	2016-04-22	2027-08-01	\$ -	Ressources Cartier inc.	
2438208	CDC	Active	32C03	37.60	2016-04-22	2027-08-01	\$ 138,521	Ressources Cartier inc.	
2438209	CDC	Active	32C03	7.25	2016-04-22	2027-08-01	\$ -	Ressources Cartier inc.	

Title No	Title Type	Title Status	SNRC	Area (ha)	Registration Date	Expiration Date	Credit	Owner	Royalties
2438210	CDC	Active	32C03	22.19	2016-04-22	2027-08-01	\$ 6,306	Ressources Cartier inc.	
2438211	CDC	Active	31N14	54.53	2016-04-22	2027-08-01	\$ 51,364	Ressources Cartier inc.	
2434329	CDC	Active	31N14	57.62	2015-10-23	2027-10-22	\$ 1,221	Ressources Cartier inc.	1% NSR to Daniel St-Pierre
2443200	CDC	Active	32C03	57.61	2016-05-05	2027-11-02	\$ -	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443201	CDC	Active	32C03	57.61	2016-05-05	2027-11-02	\$ -	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443202	CDC	Active	32C03	57.61	2016-05-05	2027-11-02	\$ -	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443203	CDC	Active	32C03	57.61	2016-05-05	2027-11-02	\$ -	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443204	CDC	Active	31N14	57.62	2016-05-05	2027-11-02	\$ 1,230	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443205	CDC	Active	31N14	57.62	2016-05-05	2027-11-02	\$ 67	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443206	CDC	Active	31N14	57.62	2016-05-05	2027-11-02	\$ -	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443207	CDC	Active	31N14	57.62	2016-05-05	2027-11-02	\$ 202	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443208	CDC	Active	31N14	57.61	2016-05-05	2027-11-02	\$ 299	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443209	CDC	Active	31N14	57.65	2016-05-05	2027-11-02	\$ 287,602	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443210	CDC	Active	31N14	57.65	2016-05-05	2027-11-02	\$ 4,961	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443211	CDC	Active	31N14	57.65	2016-05-05	2027-11-02	\$ 880	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443212	CDC	Active	31N14	57.65	2016-05-05	2027-11-02	\$ 496	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443213	CDC	Active	31N14	57.65	2016-05-05	2027-11-02	\$ 270	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443214	CDC	Active	31N14	57.65	2016-05-05	2027-11-02	\$ -	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443215	CDC	Active	31N14	57.64	2016-05-05	2027-11-02	\$ 57,568	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443216	CDC	Active	31N14	57.64	2016-05-05	2027-11-02	\$ 3,731	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443217	CDC	Active	31N14	57.64	2016-05-05	2027-11-02	\$ -	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443218	CDC	Active	31N14	57.63	2016-05-05	2027-11-02	\$ 2,099	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443219	CDC	Active	31N14	57.63	2016-05-05	2027-11-02	\$ 887	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443220	CDC	Active	31N14	57.65	2016-05-05	2027-11-02	\$ 300	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443221	CDC	Active	31N14	57.64	2016-05-05	2027-11-02	\$ 25,677	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443222	CDC	Active	31N14	57.64	2016-05-05	2027-11-02	\$ 469	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443223	CDC	Active	31N14	55.09	2016-05-05	2027-11-02	\$ 7,610	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443224	CDC	Active	31N14	50.83	2016-05-05	2027-11-02	\$ 7,653	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443225	CDC	Active	31N14	54.25	2016-05-05	2027-11-02	\$ 5,442	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443226	CDC	Active	31N14	57.34	2016-05-05	2027-11-02	\$ 25,691	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443227	CDC	Active	31N14	42.65	2016-05-05	2027-11-02	\$ 23,586	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443228	CDC	Active	31N14	6.32	2016-05-05	2027-11-02	\$ -	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443229	CDC	Active	31N14	2.46	2016-05-05	2027-11-02	\$ -	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443230	CDC	Active	31N14	9.80	2016-05-05	2027-11-02	\$ -	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443231	CDC	Active	31N14	53.22	2016-05-05	2027-11-02	\$ 3,328	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443232	CDC	Active	31N14	33.96	2016-05-05	2027-11-02	\$ 2,937	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443233	CDC	Active	31N14	50.69	2016-05-05	2027-11-02	\$ 21,286	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443234	CDC	Active	31N14	34.05	2016-05-05	2027-11-02	\$ 6,369	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443235	CDC	Active	31N14	18.80	2016-05-05	2027-11-02	\$ -	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore



Title No	Title Type	Title Status	SNRC	Area (ha)	Registration Date	Expiration Date	Credit	Owner	Royalties
2443236	CDC	Active	31N14	1.13	2016-05-05	2027-11-02	\$ -	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443237	CDC	Active	31N14	1.35	2016-05-05	2027-11-02	\$ -	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443238	CDC	Active	31N14	15.10	2016-05-05	2027-11-02	\$ -	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443239	CDC	Active	31N14	26.86	2016-05-05	2027-11-02	\$ -	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443240	CDC	Active	31N14	36.90	2016-05-05	2027-11-02	\$ -	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443241	CDC	Active	32C03	0.25	2016-05-05	2027-11-02	\$ -	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443242	CDC	Active	32C03	14.22	2016-05-05	2027-11-02	\$ 586	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2443243	CDC	Active	32C03	46.03	2016-05-05	2027-11-02	\$ 499	Ressources Cartier inc.	2% NSR to Canadian Mining House & Victor Cantore
2434769	CDC	Active	31N14	57.63	2015-11-26	2027-11-25	\$ 1,890	Ressources Cartier inc.	1% NSR to Glen Griesbach
2434770	CDC	Active	31N14	57.63	2015-11-26	2027-11-25	\$ 136	Ressources Cartier inc.	1% NSR to Glen Griesbach
2434771	CDC	Active	31N14	57.62	2015-11-26	2027-11-25	\$ 2,124	Ressources Cartier inc.	1% NSR to Glen Griesbach
2445500	CDC	Active	32C03	57.56	2016-05-25	2028-05-24	\$ -	Ressources Cartier inc.	2% NSR to Dean Boudrias
2445501	CDC	Active	32C03	57.56	2016-05-25	2028-05-24	\$ -	Ressources Cartier inc.	2% NSR to Dean Boudrias
2437912	CDC	Active	32C03	7.16	2016-04-06	2028-06-01	\$ 267,241	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437913	CDC	Active	32C03	8.11	2016-04-06	2028-06-01	\$ 617,863	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437914	CDC	Active	32C03	23.22	2016-04-06	2028-06-01	\$ 928,245	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437915	CDC	Active	32C03	7.26	2016-04-06	2028-06-01	\$ 327,288	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437862	CDC	Active	32C03	38.19	2016-04-06	2028-06-18	\$ 194,823	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437863	CDC	Active	32C03	39.43	2016-04-06	2028-06-18	\$ 884,818	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437864	CDC	Active	32C03	15.86	2016-04-06	2028-06-18	\$ 79,725	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437865	CDC	Active	31N14	3.09	2016-04-06	2028-06-18	\$ 14,069	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437866	CDC	Active	32C03	20.01	2016-04-06	2028-06-18	\$ 250,858	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437867	CDC	Active	32C03	36.26	2016-04-06	2028-06-18	\$ 1,094,424	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437868	CDC	Active	32C03	21.30	2016-04-06	2028-06-18	\$ 109,926	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437869	CDC	Active	32C03	11.28	2016-04-06	2028-06-18	\$ 58,788	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437870	CDC	Active	32C03	26.77	2016-04-06	2028-06-18	\$ 231,936	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437871	CDC	Active	31N14	5.74	2016-04-06	2028-06-18	\$ 28,829	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437872	CDC	Active	32C03	39.00	2016-04-06	2028-06-18	\$ 888,374	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2437873	CDC	Active	32C03	39.49	2016-04-06	2028-06-18	\$ 697,314	Ressources Cartier inc.	3% GMR Globex Mining Enterprises Inc.
2450535	CDC	Active	32C03	0.69	2016-06-22	2028-06-21	\$ -	Ressources Cartier inc.	1% NSR to Trip Flag Precious Metals
2450536	CDC	Active	32C03	0.35	2016-06-22	2028-06-21	\$ -	Ressources Cartier inc.	1% NSR to Trip Flag Precious Metals

4.2 Royalties and Encumbrances

Numerous mining titles comprising the Project are subject to certain agreements and royalties.

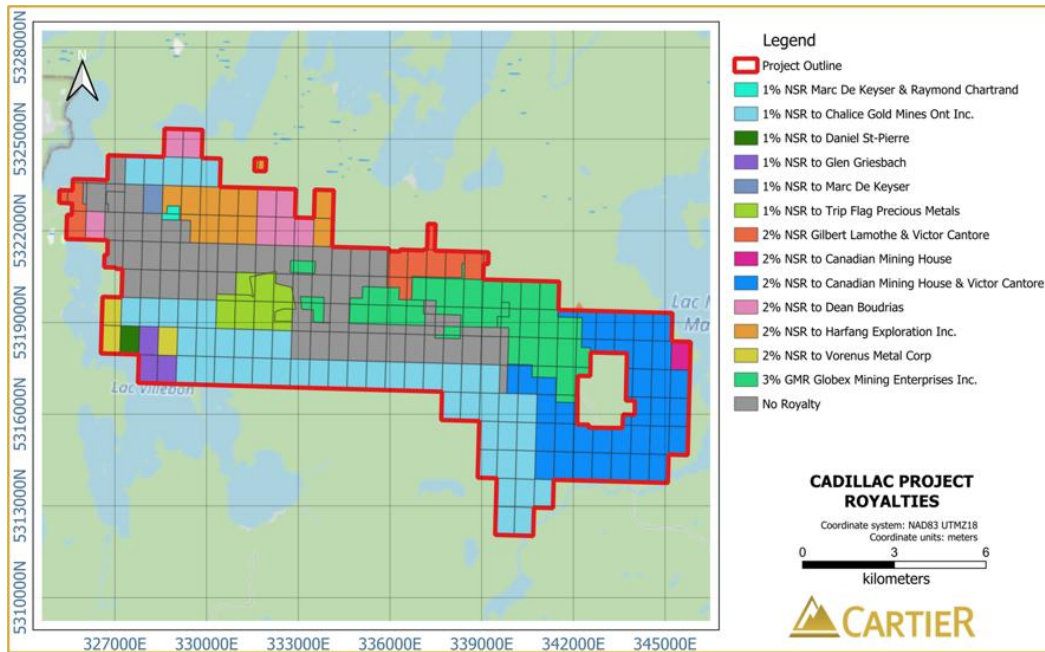
Table 4-2 summarizes the royalties held by third parties on the Cadillac Project. Figure 4-3 shows a map of the mining titles with royalties.



Table 4-2 : Royalties on the Cadillac Project

Royalty	Mining Title Count	Area (ha)
1% NSR Marc De Keyser & Raymond Chartrand	2	22
1% NSR to Chalice Gold Mines Ont Inc.	49	2,824
1% NSR to Daniel St-Pierre	1	58
1% NSR to Glen Griesbach	3	173
1% NSR to Marc De Keyser	1	58
1% NSR to Triple Flag Precious Metals	12	334
2% NSR Gilbert Lamothe & Victor Cantore	14	451
2% NSR to Canadian Mining House	1	58
2% NSR to Canadian Mining House & Victor Cantore	44	1,937
2% NSR to Dean Boudrias	8	461
2% NSR to Harfang Exploration Inc.	16	616
2% NSR to Vorenius Metal Corp.	3	173
3% GMR to Globex Mining Enterprises Inc.	54	1,500
No Royalty	73	2,933

Figure 4-3 : Location of the Royalties held by third parties on the Cadillac Project



4.3 Environmental Liabilities

There are no known environmental liabilities on the Project.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

5.1.1 Access

Access to the project is guaranteed year-round, it being easily accessible from the city of Val-d'Or by taking Trans-Canada Highway 117 east for approximately 35 km, then heading east on the Chimo gravel road for approximately 2 km.

A multitude of secondary gravel roads are available on the project and make it accessible throughout the year.

5.1.2 Climate

The regional climate is continental and typical of the Canadian Shield at this latitude, with very cold, dry winters lasting from October to April and precipitation in the form of snow. Summers are relatively short, from June to August, and hot and humid.

According to Environment Canada statistics, between 1971 and 2000, the region had an average daily temperature of 1.2°C. July has an average temperature of 17.2°C, while January drops to an average of -17.2°C. Rainfall is 635 mm per year, with 99 mm in July. Snowfall is 300 cm per year, with 61 cm in January (http://climat.meteo.gc.ca/climate_normals/).

Climatic conditions do not have a major impact on exploration activities. However, it is preferable to carry out drilling work on wetlands during the winter months.

5.2 Local Resources and Infrastructure

The nearby town of Val-d'Or has a population of approximately 32,500 according to the 2016 Canadian Census. The city was founded in the late 1930s during the gold mining boom and covers an area of approximately 3,500 km². Val-d'Or and its surrounding area currently boast some of the richest mineral deposits in Quebec.

As the largest city on the Témiscabitan road network, Val-d'Or is the junction point for most of the region's major roads. The Trans-Canada Highway (Route 117) connects Val-d'Or to the Cadillac



Project and to southern Quebec. It takes about 6 hours to drive from Val-d'Or to Montreal.

A bus service (Autobus Maheux) provides daily connections to Montreal and other cities in the region. Regular air service is operated at the Val-d'Or airport by regional airlines. Flights connect to the Montreal, Toronto, and Northern Quebec regions, among others.

A rail service (Canadian National) serves the city of Val-d'Or and offers comprehensive transportation and logistics solutions.

A 120 kV power line is provided by Hydro-Québec, ensuring a reliable network and high-quality transportation services.

5.2.1 Local Workforce

Val-d'Or is part of the regional county municipality of La Vallée-de-l'Or (MRCVO). The latter has a population of approximately 43,000 (2016 Canada Census) and covers an area of approximately 24,500 km².

A skilled and experienced workforce in exploration and mining activities is available in the region. The city also offers a wide range of services and is home to many companies specializing in the mining sector, including analytical laboratories, drilling and surveying companies, consulting and engineering firms, mining and construction contractors, and service and equipment suppliers.

5.2.2 Additional Services

The project is located near six contract concentrators in the Val-d'Or region.

Cartier's exploration office and fully equipped core shack are located in Val-d'Or, providing easy access and logistics for all exploration work carried out on the project.

5.3 Physiography

Overall, the topography is very gentle. The average elevation is around 360 to 380 m above sea level. The vegetation cover consists of moss, alder, spruce, aspen, birch, larch, and pine trees. There are numerous marshy areas in the sector, as well as sandbars.

The property has very few outcropping areas. The overburden is often thick, varying between 20 and 50 m, and consists mainly of boulders, sand, and clay.



5.4 Infrastructure

The historic mining infrastructure on the property consists of a network of 7 km of tunnels spread over 19 levels and connected by a shaft 920 m deep.

The headframe and surface facilities were dismantled in 2008, but the 25 kV power line and sand pit are still in place.



6 HISTORY

The Chimo Mine produced 379,012 ounces of gold (MERN DV 85-05 to DV-97-01) through three producers between 1964 and 1997.

Outcrops are barely visible on most of the Project. The Chimo Mine discovery was made by drilling on magnetic anomalies. Prospecting had previously revealed signs of gold, increasing the appeal of the area.

This Chapter is separated in two parts: 1) Chimo Mine Area, and 2) Area surrounding the Chimo Mine. The following text is taken from Derooff (2025) and in part from Cambior's internal report (Houle, 1995) and from Beauvais et al. (2023) and presents only the exploration work deemed relevant.

6.1 Chimo Mine Area

6.1.1 Quemartic Mines (1936 to 1938)

The Chimo mine was initially staked in the fall of 1936, then sold in September 1937 to Quemartic Mines Ltd., which transferred the project to a subsidiary, Quemartic Mines (Quebec) Ltd. During prospecting work in 1937 and 1938, visible gold was discovered at two locations in volcanic rock (zone 2, west of the mine). Two exploration drill holes totaling 336 m did not encounter any mineralization of economic interest, and the claims were eventually abandoned.

6.1.2 Chimo Gold Mines (1946 to 1947)

The Chimo Mine area was re-staked in 1943 and purchased in 1945 by Chimo Gold Mines Ltd. A magnetometric survey was conducted, followed by 45 drill holes totaling 5,800 m between 1945 and 1947. The first holes, located near the original discovery, were disappointing, but encouraging results were obtained further south, where six gold zones were soon identified (Zone 2, west of the mine). In 1948, preparatory work for the sinking of a shaft was undertaken, and the machinery was transported to the site, but work was suspended until 1963.

In 1963, a detailed magnetometric survey and an airborne electromagnetic survey were completed, followed by a 44-hole drilling program totaling 8,390 m, located 300 m east of the already known zones. Four new zones were discovered (mainly zones 2 and 3). A vertical shaft with three compartments, 183 m deep with galleries at 80, 120 and 175 m, was dug between November 1964 and June 1965. Production began on January 1, 1966, the first gold brick was cast in February, and



production ended in late August 1967, following the depletion of near-surface reserves. The ore was transported to the Bevcon mill purchased by Chimo Gold Mines.

6.1.3 Soquem and Louvem (1978 to 1989)

The Chimo Mine was acquired by Soquem in 1978, which conducted magnetic and electromagnetic surveys. A 12-hole drilling program (1,548 m) was carried out in 1978 and a second 25-hole program (6,230 m) in 1980. Soquem then transferred the mining rights to its subsidiary Louvem. Between 1981 and 1983, Louvem dewatered and rehabilitated the mine, drove exploration tunnels and completed 10,750 m of diamond drilling. Development work then began and was completed in August 1984, when production commenced. In 1984, a campaign of 33 surface drill holes discovered a new gold zone (Zone 5), 150 m south of the zones already being mined. Mining operations in the old zones were suspended to expedite the development of Zone 5. Production resumed in mid-August 1985. New drilling (29 holes for 5,755 m) was carried out in the extensions of the known zones and on geophysical targets, leading to the discovery of Zone 6 in April 1985. Successive campaigns (1986: 11 holes, 1,878 m, and 1987: 14 holes, 1,118 m) aimed to better define the known zones.

6.1.4 Cambior (1989 to 1997)

On May 8, 1989, Cambior acquired 50% of Louvem's interests and became the operator. In 1990, it purchased the remaining 50% of Louvem's interests. From 1989 to 1995, the work carried out was mainly of a construction, production, and development nature. This included deepening the shaft to 920 m with development on two additional levels (levels 18 and 19), moving the concentrator from the Lucien Béliveau plant to the Chimo mine, and constructing a paste backfill plant and an administrative office. During this period, no surface exploration campaigns were conducted, but the lateral extensions of Zone 6 were verified on Cambior's Nova property claims (1989-90 campaign: 12 holes, 2,141 m). In 1995, exploration drilling (11 holes: 3,492 m) testing the lateral extensions of Zone 5 was undertaken and led to the possible location of its extension, located approximately 750 m to the east.

6.1.5 Exploration Malartic Sud, X-Ore resources and Blue Note Mining (2001 to 2013)

On January 24, 2001, Exploration Malartic-Sud acquired 100% of the mining rights to the Chimo and Nova properties. On April 24, 2007, Exploration Malartic-Sud changed its name to X-Ore Resources,



which merged with Blue Note Mining on January 15, 2010. The first exploration work since 1997 took place in 2010, when Blue Note Mining conducted a 12-hole drilling program totaling 3,427 m to test the lateral and depth extensions of the main gold zones.

6.2 Area Surrounding the Chimo Mine

6.2.1 Raymond Tiblemont Mines (1945)

In 1944 and 1945, Raymond Tiblemont Mines drilled six holes (2B to 7B) on their Bluegrass option. One hole yielded visible gold (Ingham et al., 1945).

6.2.2 Moss Lake Development (1945)

In 1945, Moss Lake Development drilled 11 diamond drill holes (C01-C10) in the Simon West block (Gledhill, T.R. 1976). Visible gold was observed at several locations in the drill core.

6.2.3 Chimo Gold Mines (1946 to 1947)

In 1946 and 1947, Chimo Gold Mines completed a 35-hole drilling program around the Simon West showing. The zones, characterized by quartz-tourmaline veins and pyrite-pyrrhotite-arsenopyrite mineralization, were generally accompanied by significant gold grades (Hoiles, 1947).

6.2.4 Oneonta Pershing Mines (1946 to 1949)

In 1946 and 1947, Oneonta Pershing Mines conducted geological and geophysical (Mag) surveys. Eight holes were drilled in the vicinity of the Nordeau West showing. The holes encountered the first gold mineralization in the immediate area. In 1948 and 1949, Oneonta drilled 27 holes, totaling 3,400 m. Visible gold was reported in five of the holes along the iron formation (Oakley and Honsberger, 1949; Ingham, 1950a). In 1949, Oneonta intersected a horizon rich in graphitic sulfides in a hole drilled near the Bateman East showing.

6.2.5 Nordeau Mining (1957 to 1958)

In 1957, Nordeau Mining carried out a drilling program consisting of 26 holes totaling 4,688 m in the vicinity of the Nordeau East showing. Additionally, from 1957 to 1958, Nordeau Mining completed a 24-hole, 4,530 m program, which led to the discovery of gold lenses 1, 2, and 3 in the Nordeau East sector (Leclerc, 1957).



6.2.6 Vauquelin Iron Mines (1962 to 1965 and 1987 to 1994)

In 1962, Vauquelin Iron Mines drilled 14 holes totaling 1,150 m around the Nordeau West and Nordeau East sectors to test the iron potential of the iron formations (Langevin, 1962). Between 1963 and 1965, Vauquelin carried out sporadic work, including an EM survey (Dumont, 1965), and drilled five holes for a total length of 700 m (Mines de fer Vauquelin, 1965a,b). Vauquelin reacquired the property in 1987 and drilled 24 holes for 4,721 m (Nordeau West) and 30 holes for 5,889 m (Nordeau East; Beullac and Slivitzky, 1987). In 1988, Vauquelin drilled four holes totaling 1,279 m on the Nordeau West showing to test the continuation of mineralization. In 1990, Vauquelin resumed drilling to test the gold-bearing structure at depth. Of the seven holes drilled (3,471 m), five intersected the targeted gold-bearing structure. In 1990, Vauquelin drilled four holes for 1,942 m near the intersections of previous drill holes W 90-06 and W 90-07 (Nordeau West showing). From 1989 to 1990, Vauquelin expanded the geophysical coverage area around the Bateman East showing in 1988, defining additional east-southeast-trending exploration targets (Lambert and Turcotte, 1988, 1990). This was followed by a PP survey and a 15-hole (1,557 m) diamond drilling program to test a number of geophysical anomalies (Perron, 1988). In 1990, Vauquelin drilled 23 holes (3,095 m) to test the lateral extension of the interval encountered in hole BA-88-14 (Perron, 1988) and various other geophysical targets (Boulianne, 1990). This drilling defined two parallel gold-bearing zones (spaced approximately 10 m apart), which were traced over 100 m laterally and to a depth of 50 m. In 1994, Vauquelin completed a ground Mag-EM survey on the Nordeau East sector (Deragon, 1994). Six individual iron formation bands and three distinct shear zones were identified. These zones were tested in 1994 with six drill holes (619 m; Blanchet, 1994) and were in contact with iron formations and associated with quartz veins and silicified zones mineralized with massive and semi-massive sulfides (pyrrhotite, pyrite, and arsenopyrite).

6.2.7 Soquem and Louvem (1979 to 1987)

Between 1979 and 1983, Soquem conducted Mag and EM surveys (Lebel, 1979a-b; Laverdure, 1981; Glass, 1981; Blanchet and Gagnon, 1981b; St-Hilaire, 1982a; St-Hilaire, 1983b). Soquem undertook a geological compilation of previous work and the location of mineralized zones (Lebel, 1979a,c). Geological and lithogeochemical surveys followed (Blanchet and Gagnon, 1981a). They also drilled three holes in the vicinity of the Simon West showing in 1980. In 1981, following a structural mapping campaign and a compilation of work citing very good gold values from previous drilling, Soquem carried out a 9-hole drilling campaign totaling 1,311 m near the Bluegrass showing (Blanchet, 1982).



Minor sulfide mineralization was encountered, but no gold values of economic interest were noted. Soquem undertook a 41-hole, 6,640 m drilling program on the Nordeau West and Nordeau East showings (Savard and Leonard, 1980; Blanchet and Gagnon, 1981b; Blanchet, 1982). In December 1983, Louvem drilled twelve holes totaling 2,969 m near the Simon West showing (Blanchet, 1983), where numerous gold-bearing intervals were encountered. Then, in 1983, Soquem transferred its property to Louvem, completed a PP survey and drilled twelve holes totalling 2,608 m (Blanchet, 1983; Louvem, 1983). In 1984, Louvem completed 21 additional holes totaling 4,867 m on the Nordeau East sector. That same year, Louvem carried out geological mapping, geochemical (humus) and geophysical (Mag, EM, IP) surveys, and diamond drilling (5 holes, 739 m). In 1984, Louvem drilled three diamond holes (332 m; Boudreault, 1984b) to evaluate a small gold-bearing porphyry intrusion known to contain sulfides and quartz-tourmaline veins. An associated porphyry dyke, cut by arsenic-enriched faults, hosts the Marilyne gold showing. The veins and mineralization observed in the drill core were similar to those already observed at surface and in previous drilling. In the summer of 1987, Louvem began construction of an access ramp on the Simon West showing, approximately 1 km west of the Chimo mine (Rocheleau et al., 1988). This east-dipping ramp was excavated over 583 m along the contact between the Chimo volcanic belt and the sedimentary rocks of the central belt. It was designed to intersect mineralized zones #4 and #3 West at level 125 of the Chimo mine.

6.2.8 Monicor Exploration (1987 to 1988)

From 1987 to 1988, Monicor Exploration carried out surface drilling and underground exploration programs on the Simon West showing. The underground program did not confirm the extent of the mineralization (SNC Inc., 1990). In 1988, Monicor completed a four-hole (724 m) drilling campaign on the Nordeau East showing. Two types of gold mineralization were identified: 1) hosted in mafic volcanic rocks at a stratigraphic position comparable to the mineralized zones at the Chimo and Simon West mines; and 2) in sedimentary rocks to the south, associated with pyrite (Landry, 1988a). That same year, Monicor also completed a drilling program consisting of six holes totaling 1,197 m. The program identified a shear zone invaded by arsenopyrite-bearing and gold-bearing quartz (Landry, 1988b). Between January and February 1988, Monicor drilled six holes totaling 1,194 m in the area between the Nordeau West and Nordeau East showings (Landry, 1988b). These holes intersected gold zones in sequences of mafic volcanics, epiclastic sedimentary rocks, and iron formations. The iron formations are located immediately north of the mafic volcanic rocks, while the southern contact zone of the mafic rocks is characterized by shear zones and folded deformation.



6.2.9 Cambior (1987 to 1989)

In 1987, Cambior conducted a regional VLF-EM and Mag survey covering the area near the Larder Lake-Cadillac fault zone from east of Val-d'Or to Machi-Manitou Lake (approximately 40 km) and identified numerous anomalies, including strong magnetic trends underlying much of the east-west extent of the East Cadillac property (Podolsky, 1987). In 1988, Cambior conducted a comprehensive reverse circulation ("RC") drilling program in the overburden and heavy mineral geochemical sampling (MacNeil and Averill, 1988) south of the Chimo mine. According to the author of the report, the RC drilling results indicate that Pontiac Group rocks, dominated by turbidites, underlie the property, rather than Trivio Group rocks, as was generally believed. This interpretation dictated that the Larder Lake-Cadillac fault zone, a regional structure separating the Pontiac Group and the Trivio Group, must pass through the northern part of the property, and not through the south, as some workers had previously suggested. In 1989, Cambior reported the results of a 16-hole diamond drilling program totaling 5,458 m (Lortie, 1989). These drill holes targeted anomalous gold concentrations in glacial overburden. A total of 1,359 core samples with a cumulative interval length of 1,959 m were collected and analyzed for gold content. Sulfide mineralization (trace to 5%), consisting mainly of pyrite, arsenopyrite, pyrrhotite, and chalcopyrite, was found in association with quartz and carbonate veins and veinlets in drill holes southwest of the Chimo mine deposit.

6.2.10 Plato Gold (2006 to 2011)

Plato Gold completed a 7,363 m drilling program between October 2006 and March 2007. The objective of this initial program was to drill the Nordeau East and Bateman East showings for the first time and to identify the best targets for future exploration (Bourgoin and Castonguay, 2007). In December 2007, Plato compiled a detailed digital compilation of all historical exploration results and provided recommendations for further exploration. Between January and September 2008, Plato completed a 14-hole (8,555 m) diamond drilling program on the Nordeau West showing, successfully intersecting the main zone to a depth of 700 m and demonstrating good grade and continuity. From October to December 2009, Plato completed 4,699 m of diamond drilling in the Nordeau East area (11 holes) and 834 m in three holes in the Bateman East area. The drilling program was designed to test the longitudinal continuation of mineralization previously identified in the Nordeau West and Nordeau East showings (Kromo and Langton, 2010). Between May and June 2010, Plato completed a three-hole drilling campaign totaling 836 m on Nordeau East (Langton and Pacheco, 2011). From January to May 2011, Plato completed 27 drill holes totaling 11,966 m, of



which 8,758 m (17 drill holes) were completed on Nordeau East and 3,208 m (10 drill holes) on Bateman East.

6.2.11 Globex Mining (2014)

In June 2014, Globex conducted a sampling campaign on drill cores from work carried out by Plato between 2006 and 2011. The purpose of this sampling was to test potentially mineralized zones that had not been sampled previously and to verify the feasibility of tracing mineralized zones through unsampled areas near gold zones (Manon and Pierre, 2014). Most of the sampling focused on drill cores from the Nordeau West sector, where 1,198 samples (1,483 m) from 22 drill holes were collected and reanalyzed; 45 drill core samples (53 m) were collected from three holes drilled in the Nordeau East sector, and an additional 27 samples (33 m) were collected from three holes in the Bateman East sector.

6.2.12 Chalice Gold Mines (2016 to 2019)

In October 2016, Chalice Gold Mines conducted a wide range of surface exploration programs and drilling campaigns. The work resulted in the creation of a comprehensive geochemical database to better define potential exploration targets. Components included extensive soil sampling campaigns (MMI; 2016, 2017, and 2018), biogeochemical sampling campaigns (black spruce bark; 2017 and 2018); regional rock mapping/sampling surveys (2016, 2017, and 2018), and finally, sampling of selected drill cores from 26 historic drill holes. In addition, the claims were covered by ground-based (OreVision) and airborne (Mag, VLF-EM) geophysical surveys and an airborne LiDAR survey. Details of the various exploration programs are described in Scott et al. (2017a), Dallmeier et al. (2018) and Letwinetz et al. (2018a,b).

Between September 2017 and March 2019, Chalice drilled 97 diamond holes totaling 33,632 m. The programs tested gold targets along the Larder Lake-Cadillac fault zone, around the Simon West and Nordeau West showings, as well as numerous geochemical and geophysical targets identified by Chalice's previous surveys (Simon West, Far Simon West, North Contact, Lac Rapides, Anderson, and Legrand areas; Scott et al., 2017, Cornick et al., 2019a).

6.2.13 O3 Mining (2020 to 2022)

Between 2020 and 2021, O3 completed a 20,889 m (46 holes) drilling program to follow up on promising historical drill intersections in the Simon West, Nordeau West, Nordeau East, and North





Contact areas (Ballesteros, 2021). A stripping campaign followed by channel sampling, as well as geophysical and soil sampling surveys, were also carried out (Ballesteros, 2021 and 2022). Soil samples covered an area between the North Contact and Nordeau East showings and another area adjacent to the Big Bend showing. Manual stripping and channel sampling were carried out near the North Contact and Big Bend showings. The purpose of this stripping was to validate the geometry of the gold-bearing quartz vein system. The stripped areas on Big Bend exposed the iron formation of the Trivio structural complex. The geophysical surveys conducted by O3 consisted of Mag (116 km) and PP (57 km) surveys northeast of Matchi-Manitou Lake. Ten new drill targets were identified (Simard, 2021).



7 GEOLOGICAL SETTING AND MINERALIZATION

The Cadillac Project is located in the Archean Province of Superior, most specifically in the southeastern part of the Abitibi Subprovince, commonly known as the Abitibi Greenstone Belt (AGB). The following text is taken from Deroff (2025).

7.1 Archean Superior Province

The Archean Superior Province forms the heart of the North American continent and is surrounded by Paleoproterozoic provinces to the west, north, and east, and by the Mesoproterozoic Grenville Province to the southeast (Figure 7-1).

Figure 7-1 : Location map of the Abitibi greenstone belt (green) within the Archean Superior Province (Grey).
After Monecke et al. (2017).



Tectonic stability has prevailed for approximately 2.6 Ga in large parts of the Superior Province. Proterozoic and younger activity is limited to margin rifting, the emplacement of numerous mafic dyke swarms (Buchan and Ernst, 2004), reactivation by compression, large-scale rotation at about 1.9 Ga, and rifting failure at about 1.1 Ga. With the exception of the northwest and northeast margins



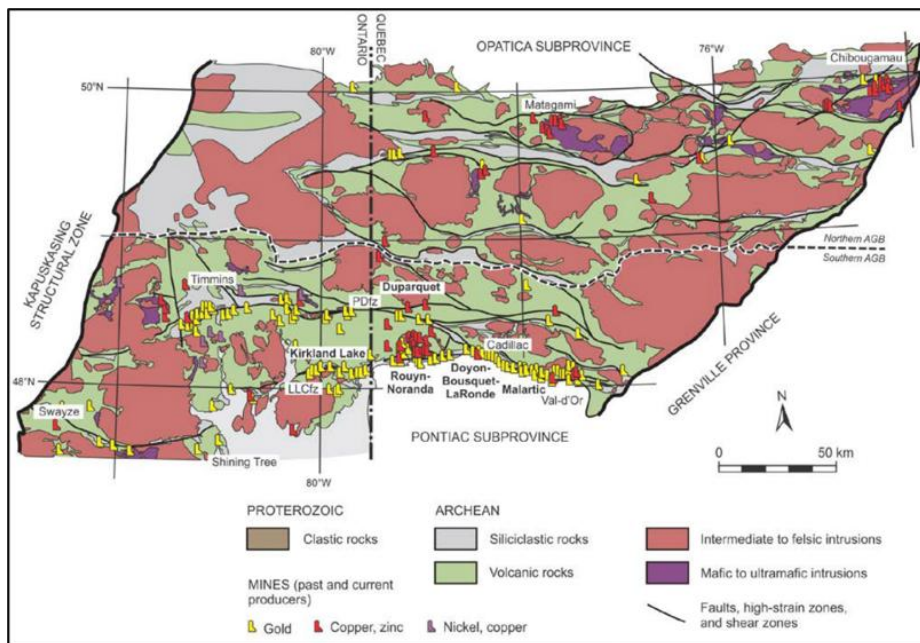
of the Superior Province, which were pervasive deformed and metamorphosed between 1.9 and 1.8 Ga, the craton escaped ductile deformation.

A key feature of the Superior Province is its linear subprovinces (or “terrane”), with distinctive lithological and structural characteristics, accentuated by subparallel faults (Card and Ciesielski, 1986). Orientations are generally east-west in the south, west-northwest in the northwest, and northwest in the northeast. The term “terrane” is used to mean a geological domain with a distinct geological history prior to its merger into the Superior Province during the 2.72 to 2.68 Ga assembly events. The term “superterrane” indicates evidence of internal fusion of terranes prior to Neoproterozoic assembly. The term “domain” is defined as a distinct region within a terrane or superterrane.

7.1.1 Abitibi Subprovince

The Abitibi Subprovince is located in the southern part of the Superior Province (Figure 7-2).

Figure 7-2 : Geological map of the Abitibi Greenstone Belt (Monecke et al., 2017).



The Abitibi Greenstone Belt (AGB) is bounded on the west by the Kapuskasing structural zone and on the east by the Grenville Province. To the north, the Abitibi Subprovince is in structural contact with the Opatika Plutonic Subprovince. The southern boundary of the Abitibi greenstone belt is marked by the Larder Lake-Cadillac Deformation Zone (LLCDZ), a major structural feature marking the

contact with the younger metasedimentary rocks of the Pontiac Subprovince.

The AGB was formed over a period of approximately 150 Ma. It consists of east-trending synclines of largely volcanic rocks and intermediate domes, cut by synvolcanic and/or syntectonic plutonic rocks (gabbro-diorite, tonalite, and granite composition) and alternating with east-trending bands of turbiditic wackes (Ayer et al., 2002; Daigneault et al., 2004; Goutier and Melançon, 2007; Monecke et al., 2017). Most of the volcanic and sedimentary layers have a subvertical dip.

The volcanic and sedimentary assemblages are generally separated by steep east-trending faults. Some of these faults, such as the LLCDDZ and the Porcupine-Destor Deformation Zone (PDDZ) and similar structures in northern Abitibi, cut across the entire belt and show evidence of superimposed deformation events (early thrusting, late detachment, and extension (Daigneault et al., 2004; Benn and Peschler, 2005; Bateman et al., 2008).

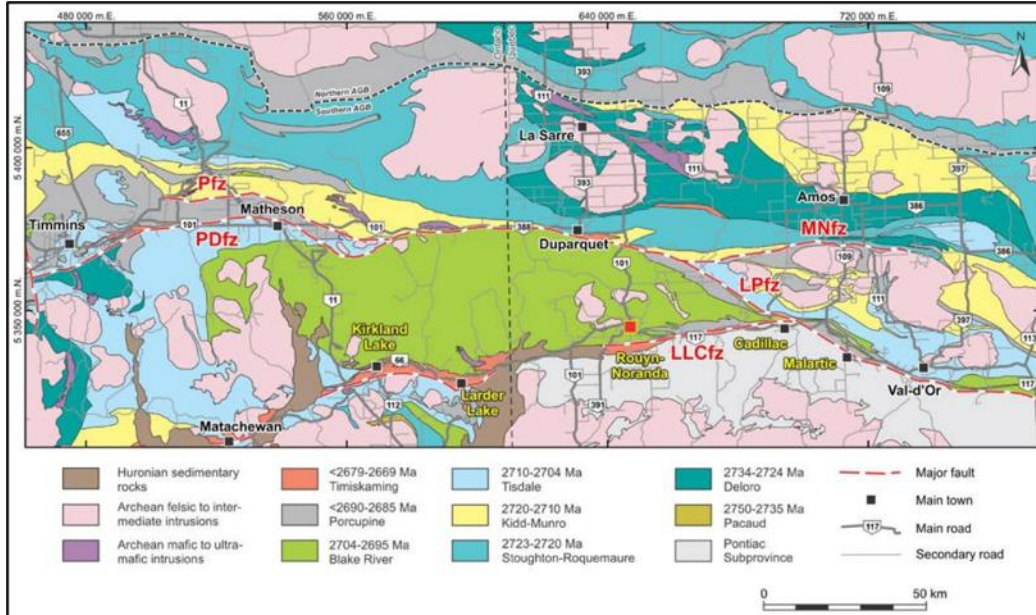
The LLCDDZ is transcrustal and inherited from the accretionary suture between the Pontiac and the AGB. The fault is important not only for its metallogenic richness, but also for its geodynamic models and its juxtaposition of varied lithological assemblages along its subsidiary faults. When the E-W and ESE-W-NE segments of the fault cross the AGB, they reflect a deep asymmetry, a feature that has influenced the styles and episodes of gold mineralization.

In addition, the AGB is cut by numerous late tectonic plutons ranging in composition from gabbro to granite with smaller dykes or intrusions of syenite, lamprophyre, and carbonatite. The greenstone belt is affected by widespread greenstone facies metamorphism (Jolly, 1978; Dimroth et al., 1983; Powell et al., 1993; Benn et al., 1994; Faure, 2015). The degree of metamorphism increases to amphibolite facies at the periphery of certain plutons and as one approaches the Pontiac and Opatica Subprovinces or the Grenville Proterozoic Province.

According to Monecke et al., 2017, the AGB is subdivided into eight discrete episodes or stratigraphic assemblages (Figure 7-3), based on U-Pb zircon age groupings. Submarine volcanism occurred mainly between 2795 and 2695 Ma and was followed by sedimentation in large deep basins, then by large-scale overlapping and tight folding. New U-Pb zircon ages and recent mapping by the Ontario Geological Survey and Géologie Québec clearly show a similarity in the timing of volcanic episodes and ages of plutonic activity between the north and south of the AGB.



Figure 7-3 : Geological map of the southern Abitibi greenstone belt (Monecke et al., 2017).



Two ages of discontinuous sedimentary basins are recognized: early, large, laterally extended Porcupine-type basins consisting of fine-grained clastic rocks (turbidites), followed by Timiskaming-type basins composed mainly of coarser clastic rocks and small amounts of volcanic rocks, which are largely proximal to major faults where strike-slip movements have occurred (Thurston and Chivers, 1990; Mueller et al., 1992; Ayer et al., 2002; Goutier and Melançon, 2007).

The assembly groups, from oldest to most recent, are:

- Pacaud Assemblage (2750-2735 Ma);
- Deloro Assemblage (2734-2724 Ma);
- Stoughton-Roquemaure Assemblage (2723-2720 Ma);
- Kidd-Munro Assemblage (2720-2710 Ma);
- Tisdale Assemblage (2710-2704 Ma);
- Blake River Assemblage (2704-2695 Ma);
- Porcupine Assemblage (<2690-2685 Ma);
- Timiskaming Assemblage (<2679-2669 Ma).

Three major intrusions are found in the local stratigraphy: 1) the Bourlamaque batholith (Campiglio, 1977), a synvolcanic intrusion of coarse-grained quartz-diorite (2700 ± 1 Ma) of transitional affinity, which is interpreted as the magma chamber that fed the volcanism of the Val-d'Or Formation; 2) the



Bevcon Pluton, a similar but more differentiated fine- to medium-grained tonalite of transitional affinity; and 3) the East Sullivan Stock, a post-kinematic alkaline monzonitic stock (2684 ± 1 Ma; Taner, 1996). Numerous calc-alkaline tonalitic to granodioritic intrusions, subconcordant to discordant subvolcanic to post-kinematic sickers, and a series of early to post-kinematic quartz-feldspar dykes occur throughout the region.

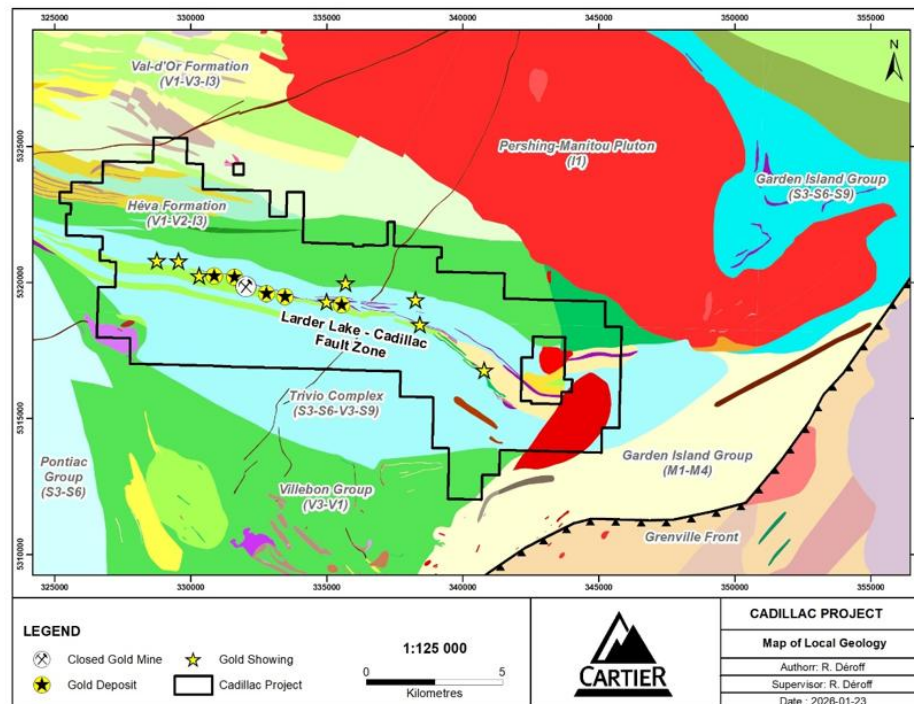
7.2 Local Geology

The following text is taken from Deroff (2025) and in part from Sauvé et al. (1988), Rocheleau et al. (1997), Moorhead et al. (2000), and Plouffe (1990).

The Cadillac Project is located in the Southern Volcanic Zone of the Abitibi Subprovince. This zone is crossed in the north by the possible extension of the Larder Lake-Cadillac deformation zone.

Figure 7-4 shows the local geology of the Cadillac Project.

Figure 7-4 : Local Geology of the Cadillac Project



7.2.1 Lithologies

The project is crossed by the volcano-sedimentary rocks of the Trivio Complex. According to the work



of Racine (1989) and Rocheleau et al. (1997), the Trivio Complex is composed of a structural complex containing a set of lenticular bands of sedimentary rocks and mafic volcanic rocks, in shear contact with each other, without it being possible to link them stratigraphically.

Sedimentary rocks form a rhythmic sequence of proximal turbidites, composed of fine quartz-feldspar sandstones and siltstones, interbedded with levels of magnetite-bearing iron formations, coarse feldspar sandstones, and a few levels of petromict conglomerates.

Volcanic rocks are grouped within two main belts (north and south volcanic belts) and consist mainly of massive, pillow, and, more rarely, brecciated basaltic and andesitic flows (Racine, 1989; Rocheleau et al., 1997). Volcaniclastic and exhalative rock horizons interdigitate with sedimentary rocks, indicating that volcanism was active during sedimentation.

7.2.1.1 Sedimentary Units

The most common sedimentary facies is a rhythmic sequence of fine quartz-feldspar sandstones and siltstones in beds of varying thickness, averaging 5 to 10 cm, often showing normal grain sorting, alternating with thin interbeds of shales 1 to 5 cm thick. Sometimes, a thin layer of parallel laminations is observed at the top of the beds. In thin sections, these sandstones consist of rounded feldspar and quartz fragments. Rock fragments are rarely observed and appear to be restricted to the coarser beds. The abundant matrix (30 to 40%) is completely recrystallized into quartz, muscovite, biotite, and chlorite.

A magnetite-bearing iron formation can be traced on federal magnetic maps and in drill holes over a distance of more than 16 km from the former Chimo mine to the Matchi-Manitou Lake area (Rocheleau et al., 1988). The magnetite beds are interbedded with granoclassed sandstone-pelitic beds and form a folded band varying in width from 3 to 70 m in the Chimo mine areas. The iron-bearing layers are characterized by alternating millimetric to centimetric laminations rich in magnetite, white chert laminations, and green beds of iron silicates with or without magnetite. Under the microscope, the magnetite appears as small irregular grains 0.05 mm in diameter or as subidiomorphic poeciloblastic grains 0.3 mm in diameter. The quartz forms a mosaic of 0.05 mm grains and is likely to be recrystallized chert (Sauvé et al., 1987). The iron silicates include grunerite, ferrohornblende, chlorite, and a small amount of biotite. The presence of slightly manganese-bearing almandine garnet has been noted in the southern iron-bearing level of the Chimo mine (Sauvé et al., 1987).

A second sandstone facies consists of coarse, conglomeratic feldspathic sandstone (2 to 4 mm),



chloritized and poor in quartz (< 5%). Outcrop observations show beds having an average thickness of 15 to 30 cm. The boundaries between the beds are poorly defined. These sandstones are generally massive and lack sedimentary structure. Thin interbeds of shale break up the monotony of this sequence. In thin sections, the coarse sandstones consist of plagioclase (albite) fragments in a matrix similar to that of the sandstones described above. The matrix is recrystallized into chlorite, biotite, and quartz, in addition to containing 3 to 7% carbonates (calcite).

The petromict conglomerate with pebbles, cobbles, and blocks, generally highly deformed, occurs in lenticular beds of variable thickness, massive or grain-classified. The percentages of fragments and matrix are particularly difficult to determine because of the deformation. However, in the least deformed areas, the matrix appears to be sufficiently abundant to support the fragments. The fragments are composed mainly of blocks of volcanic origin, primarily tuffs and felsic crystal tuffs, blocks of intrusive origin of tonalitic composition and, in smaller quantities, black chert pebbles and mafic volcanic rocks. More rarely, it is possible to identify pebbles of felsic volcanic rocks, sedimentary rocks, and fuchsite. The matrix of these conglomerates is either sandy or silty and has the same composition as the facies of fine quartz-feldspar sandstones and siltstones. These lenticular conglomerate levels are interpreted as being the filling of submarine channels.

7.2.1.2 Volcanic Units

Mafic and intermediate volcanics, mainly basalts and andesites, have a massive, pillow-like, and more rarely brecciated facies. Massive volcanics are generally aphanitic, although locally granular (1 to 2 mm). Pillow basalts are sometimes vesicular and have pillows of highly variable dimensions with very little associated hyaloclast material. Volcanic breccias are uncommon and generally restricted to thin lenticular horizons. These are generally flow breccias with volcanic fragments, more or less abundant (0 to 60%) and small in size (1 to 5 cm), in a hyaloclastite matrix. The pyroclastic rocks consist of mafic, highly feldspathic, ash, crystal, lapilli, and block tuffs. Certain levels particularly rich in graphite are found in the vicinity of the Chimo mine. In thin sections, the mineralogy observed is similar for all volcanic rocks. The mineralogical assemblage and textures observed are essentially metamorphic, with the rocks being completely recrystallized. The major constituents are quartz, chlorite, actinolite, and epidote (zoisite and clinozoisite). There is a minimal amount of biotite and opaque minerals (magnetite, ilmenite, and pyrite) and traces of sphene, leucoxene, and tourmaline.



7.2.2 Lithogeochemistry

Some analyses of mafic volcanics indicate high MgO contents (10.0%, 13.4%, and 17.1%), consistent with the compositions of magnesian basalts (Rocheleau et al., 1997). Basalts and andesites have Zr/Y ratios ranging from 2 to 6. They are grouped into two trends: the first, with a Zr/Y ratio of 2.5, would be tholeiitic, and the second, less abundant, with a Zr/Y ratio of 5.5, would be transitional. Volcaniclastic rocks have more felsic compositions (SiO₂ between 53 and 73%), generally dacitic (Rocheleau et al., 1997). They have Zr/Y ratios varying between 2 and 12.

According to Racine's study (1989), the northern volcanic belt of the Trivio Complex is composed mainly of tholeiitic basalts, while the volcanics of the southern belt are mainly tholeiitic andesites. The lavas have Zr/Y ratios ranging from 2.2 to 5.5; the vast majority of the volcanics analyzed have ratios lower than 4, clearly indicating their tholeiitic affinity. The volcanics from both belts form a straight line on a Zr-Y diagram, indicating that they are probably comagmatic (Racine, 1989).

7.2.3 Hydrothermal Alteration

At the former Chimo mine, mineralized zones and their margins are characterized by hydrothermal alteration that manifests itself in different ways depending on the lithology:

- Silicification of the host rock occurs in sediments associated with iron formations. This process, which is generally accompanied by chloritization and biotization, is limited to the immediate outcrop of the mineralized zones.
- Carbonation affects a much larger volume of rock in volcanic or pyroclastic lithologies. It is characterized mainly by calcite, but also by chlorite.
- Tourmalinization is present in all zones, but its distribution is very uneven. It can be found locally at the margins and in mineralized veins.
- Sulfurization manifests itself in the presence of granular arsenopyrite and pyrrhotite in veins or semi-massive horizons or in disseminated form in quartz veins and in the cap rocks. Arsenopyrite replaces pyrite and pyrrhotite.

7.2.4 Metamorphism

In the Trivio Complex, a garnet-hornblende-calcite-quartz ± biotite association is linked to WNW-ESE faults delimiting contrasting blocks (volcanics and sediments) and marking areas with a steep gradient. The northern zone contains pyrrhotite-rich metasomatite lenses with low gold and copper



grades. These lenses are oriented N260 to N280°, with dips of 62 to 75° to the north. Their thickness varies from 0.5 to 3 m. The southern zone extends along the axis of the former Chimo mine and the Nordeau showing and corresponds to the contact between a block of volcanic rocks and a band of iron formation. In this zone, which contains economically viable gold mineralization, the garnet-hornblende-calcite-quartz association is accompanied by pyrrhotite, arsenopyrite, chalcopyrite, and pyrite.

Berclaz (1993) conducted a mineralogical and geochemical study of amphibole- and garnet-rich metasomatized rocks located within the volcanic belts of the Trivio Complex. The recrystallization of volcanic and sedimentary rocks into massive amphibolites and garnet-bearing alumina schists took place in the high-temperature domain of a hydrothermal system with a high geothermal gradient and is attributed to a hydrothermal episode prior to the main Kenorian phase of deformation and metamorphism. The pressure-temperature conditions for this high-temperature metasomatic episode are estimated at 600-700 MPa and 500-580°C (Berclaz, 1993). This event is characterized by calcic, ferruginous, and aluminous hydrothermal metamorphism, including almandine garnet, calcic amphibole, calcic plagioclase, biotite, chlorite, and pyrrhotite (Berclaz, 1993).

7.2.5 Structure

The Trivio complex forms a structural discontinuity and represents an area of intense deformation that cuts through terrain with less deformation. This lithotectonic domain is characterized by the presence of anastomosed deformation corridors (shear zones and/or brittle faults) of varying widths (metric, decametric, and even hectometric), which delimit structural blocks measuring hundreds of meters to kilometers in size. The intensity of deformation (flattening and stretching of geological materials), which is relatively low in the center of the structural blocks, increases significantly as it approaches the deformation corridors, where the rock is completely mylonitized. The transition from slightly deformed rocks to mylonitized rocks is either gradual (over tens of meters) or abrupt (over a few meters).

Regardless of the width of the deformation corridors, all structural elements (folds, schistosity, lineations, etc.) increase in intensity in the mylonitic zones and become parallel, in some cases forming tectonic bedding. The rock is then transformed into chlorite and carbonate schist or amphibolite. The schistosity and shear planes, generally E-W with a steep dip to the north or south, are then defined by the alignment of phyllosilicates, mainly biotite and chlorite, and amphiboles. As



indicated by Racine (1989) for the Larder Lake-Cadillac deformation zone, schistosity sometimes forms a slight angle with the shear, allowing a dextral strike-slip component to be defined at the surface. The stretch lineation is generally very well developed and always very steep, dipping either eastward or westward.

Within the less deformed structural blocks of these tectonic zones, regional E-W folds can be recognized over a distance of several kilometers. These folds have been mapped in the sedimentary and volcanic rocks of the Trivio Complex.

Two major features of Archean deformation evolution have been documented on the Chimo Mine project:

- Regional pre-deformation phase: characterized by early, isoclinal, highly localized folds. There is also an S1 schistosity, which is difficult to recognize due to the predominant nature of the D2 regional deformation. This S1 schistosity, identified locally only in sedimentary and pyroclastic rocks but absent in basalts, forms an angle of 10 to 15° with the S2 schistosity that overlaps it (Racine, 1989).
- Regional deformation phase: responsible for the E-W tectonic grain, the main P2 folds, and the dominant S2 schistosity. Interpreted as a flow schistosity, it represents a structural event that is definitely more significant than the other phase of Archean deformation. This main phase is associated with very tight mesoscopic folds, with an isoclinal tendency, generally oriented E-W, with a variable dip but generally towards the NE, from 30 to 70°. Shears sub-parallel to the axial planes of these mesoscopic folds are also considered to be contemporary with this deformation phase. The S2 schistosity is sub-parallel to the axial planes of most P2 mesoscopic folds, to E-W shears and also to the planes of flattening of geological objects (pillows, vesicles, fragments, crystals). Microscopically, this foliation results from the preferential alignment of all the rock's mineralogy, particularly metamorphic minerals such as chlorite, sericite, biotite, and certain amphiboles.

7.2.6 Stratigraphic Relationships

No U-Pb isotopic dating has been performed on the sedimentary rocks of the Trivio Complex. However, this complex is comparable to other predominantly sedimentary lithotectonic domains in the southern part of the Abitibi Subprovince, west of the Project. Detrital zircons from these assemblages have been dated using the U-Pb method and indicate the maximum age of these sedimentary rocks. These assemblages include the Kewagama Group (< 2687 Ma; Davis, 1991), the



Cadillac Group (< 2688 Ma; Davis, 1991) and the Pontiac Group (< 2688 Ma; Davis, 1991). Furthermore, to the west, the Trivio Complex is cartographically continuous with the Pontiac Group. The age of the Trivio sediments should prove to be comparable to that of the Pontiac.

The volcanic rocks of the Trivio Complex form lenticular bands within a structural complex dominated by sedimentary rocks. Where they have been documented, the contacts between the volcanics and the encasing sedimentary rocks are sheared (Rocheleau et al., 1997). There appear to be no stratigraphic relationships between the sedimentary and volcanic rocks. Although the Trivio volcanic rocks are allochthonous with respect to the sedimentary rocks, they may have originated from adjacent volcanic complexes and been interbedded by thrust faults in the Trivio sedimentary assemblage. The mafic volcanics are mainly composed of tholeiitic basalts and andesites and, to a lesser extent, transitional basalts and andesites. These tholeiitic lavas are similar to those of the lower part of the Val-d'Or Formation or the Jacola Formation. The presence of mafic basalts locally intercalated in the mafic lava bands of the Trivio (Rocheleau et al., 1997) supports the hypothesis that a portion of the tholeiitic mafic lavas can be correlated with the Jacola Formation. The basalts and andesites with transitional affinity are similar to the mafic lavas of the Val-d'Or Formation. The intermediate to felsic volcanoclastics are of transitional to calc-alkaline affinity. They could be correlative to the intermediate to felsic volcanoclastics of the Val-d'Or Formation.

Therefore, it is plausible that the lenticular bands of volcanic rock in the Trivio Complex represent sections of the Val-d'Or Formation and, to a lesser extent, the Jacola Formation, which were thrust southward and interbedded into the sedimentary assemblage of the Trivio Complex during compressive deformation.

West of Trivio Lake, the sheared basaltic and ultramafic lavas of the Piché Group form a horizon enclosed within the Larder Lake-Cadillac deformation zone, at the contact between the Cadillac and Pontiac sedimentary groups. If the same hypothesis used for the lava horizons in the Trivio Complex is applied to the Piché Group, the latter's mafic and ultramafic lavas could represent structural scales of mafic-ultramafic volcanic units of the Val-d'Or Formation, possibly the Jacola, Dubuisson, or La Motte-Vassan Formations, overridden to the south and interbedded in deformed sedimentary units.

7.2.7 Mineralization

Two gold deposits (the historic Chimo Mine and the West Nordeau occurrence) and several occurrences are known on the Project.



The Chimo mine produced 379,012 ounces of gold (MERN DV 85-05 to DV-97-01) through three producers between 1964 and 1997. The West Nordeau deposit lies 1.5 km east of the Chimo Mine. Gold mineralization in the area is typically concentrated along the LLCFZ and related secondary structures. This is evident at the closed Chimo Mine and the West Nordeau deposit, where gold occurs with quartz and arsenopyrite in longitudinal high-strain (“shear”) zones within the mafic volcanic rocks and in bands of semi-massive arsenopyrite and pyrrhotite associated with banded magnetite iron formation units (Sauvé et al., 1987).

Gold mineralization on the Project occurs epigenetically in silicified lodes with disseminated sulphides, spatially related to banded iron formation and altered shear zones with temporally related quartz $\pm$ carbonate veins. When related to shear zones, Au mineralization typically occurs in volcanic units with disseminated arsenopyrite, pyrite and chalcopyrite. Graphite horizons are also common with this type of mineralization.

In addition, several new prospective gold zones have been discovered in recent years, namely the South Nordeau, North Contact, Far West Simon and Lac Rapides zones, all of which are roughly parallel to, and closely associated with, the LLCFZ.

7.2.7.1 Chimo Mine Deposit

The following information is taken from Sauvé et al. (1988), Plouffe (1990), Langton and Jourdain (2019) and Savard and D’Amours (2020).

Mineralization at the Chimo Mine deposit consists of five main mineralized structures (zones 1 to 3 and 5 to 6).

Zone 1 generally follows the northernmost iron formation. Mineralization consists mainly of a juxtaposition of centimetric to decimetric veinlets of coarse arsenopyrite. Semi-massive layers of pyrrhotite with minor pyrite are sometimes present, but these sulphides are almost always barren if not accompanied by arsenopyrite. Lenses or veins of white quartz are found along the margins of uneconomic sulphidized areas. Arsenopyrite veins are sometimes deformed into small tight folds and occasionally intersect the bedding at a low angle.

Zone 2 roughly follows the contact between the southernmost iron formation layer and a mafic intrusion. Sulphide minerals are found in a brownish-coloured area rich in biotite. Pyrrhotite and coarse arsenopyrite define thick ribbons parallel to schistosity. The margins of mineralized areas consist of finely ribboned pyrrhotite or disseminated arsenopyrite. Bluish quartz forms irregular lenses and veins and contains visible gold but few sulphides.



Zone 3 is located along the southern contact of the schistose and carbonatized mafic intrusion. Mineralization consists, on average, of 3 to 5% disseminated sulphides and multiple veins of bluish quartz with good continuity. The veins show ribboning that may include layers of wallrock containing disseminated sulphides. Sulphides consist mainly of fine arsenopyrite with lesser pyrrhotite and small amounts of pyrite and chalcopyrite. Quartz veins contain plenty of visible gold, especially near their borders, but few sulphides.

Zone 5 lies within a schistose and carbonatized greyish lava and includes thin units of graphitic schist. The sheared zone may have followed a thin layer of tuff, but deformation and alteration extended into the surrounding volcanic rocks. In the upper parts of level 5, the main type of mineralization is a large concordant lens of black quartz intersected by numerous graphitic veins. Sulphide content is low, but visible gold is present. At each end, the lens may terminate abruptly against graphitic schist. Elsewhere, the lens splits into parallel quartz veins that locally intersect with each other or intersect schistosity. These multiple digitations or veins give the impression that the wide lens formed gradually by accretion in a dilating zone. The rock encasing the lens of graphitic quartz contains many quartz veins and veinlets that contain only small amounts of arsenopyrite, although greater concentrations are found along the selvages. These veins of quartz and arsenopyrite do not contain graphite. They yield good gold grades but are narrow and lenticular. In the lower parts of level 5, the graphitic quartz lens disappears and gives way to graphite-free quartz veins with disseminated arsenopyrite rims.

Zone 6 is similar in many ways to Zone 5. It consists of several subparallel en-echelon gold-bearing lenses. These lenses are composed of sheared and carbonatized rock of volcanic and pyroclastic origin, injected with multiple lenticular quartz veins. Disseminated arsenopyrite, pyrrhotite and pyrite occur within the veins and along the vein walls. Free gold is found in quartz veins and as scattered fine grains in the selvages or as inclusions in arsenopyrite. Layers of graphitic schist occur in the eastern part of the zone.

Gold appears to be the last metal phase in the mineralized zones of the former Chimo mine. It is preferentially associated with arsenopyrite and some of the quartz veins. The layers of semi-massive coarse arsenopyrite have consistently elevated gold grades; however, the grade is much lower when arsenopyrite is disseminated. Massive pyrrhotite and pyrite have negligible gold grades when arsenopyrite is absent.

Quartz veins rimmed by disseminated arsenopyrite have good gold grades. Visible gold is present within the veins and along the vein walls. Veins devoid of arsenopyrite have negligible gold grades,



except the graphitic quartz veins containing visible gold.

Based on the nature of the host rock in the Chimo Mine deposit, the mineralized zones have been divided into two types of lode deposits: semi-massive sulphide veins associated with iron formations (zones 1 and 2) and lenticular quartz veins associated with altered volcanic rocks mineralized with arsenopyrite (zones 3, 5 and 6).

These two types of gold mineralization are unlikely to be genetically distinct because they occur together in the Chimo Mine and are likely the distinct manifestations of mineralization in different host rocks. Mineralization is associated with injections of quartz and sulphide minerals into sheared zones at the contacts between lithologies of contrasting competency and chemical composition.

These contacts promoted the development of deformation and micro-porosity in ductile rock and fracturing and brecciation in more competent rock, thereby promoting the circulation of hydrothermal fluids.

At the former Chimo mine, mineralized zones are accompanied by a variety of hydrothermal alteration types, depending on the host lithology:

- Silicification took place in sedimentary host rocks where iron formations were present. Typically accompanied by chloritization and biotization, it is limited to the immediate wallrock in mineralized areas;
- Carbonatization affects a much larger volume of rock in volcanic or pyroclastic lithologies. Primarily characterized by calcite, chlorite may also be present;
- Tourmalinization affects all the zones, but its distribution is very irregular. It can be found locally along the margins of mineralized veins; and
- Sulphidation manifests as pyrrhotite and coarse arsenopyrite in veins, semi-massive horizons, or disseminations in quartz veins and along vein walls. Arsenopyrite replaces pyrite and pyrrhotite.

7.2.7.2 West Nordeau

Gold mineralization on the claims in the West Nordeau area occurs in four distinct lenses or zones, known from north to south as North Zone, Zone 1, Zone 2 and Zone 3, in shear zones that transect the mafic volcanic rocks of the Trivio Group. Wall rocks are massive to pillowed or brecciated basalts and andesites with sporadic tuffaceous horizons and minor graphitic schist. Common alteration types in the sheared rocks include silicification, amphibolitization, carbonatization and biotitization. Gold is found in brecciated zones cemented with grey/smoky quartz lodes, veins or veinlets with



brown tourmaline, carbonates and sulphides in an “en échelon” pattern within the wider deformation corridors (Jean, 1990). Arsenopyrite is the major sulphide constituent (3-15%) with some amounts of pyrite, pyrrhotite and traces of chalcopyrite. Gold is locally found as free grains intergrown with arsenopyrite.

All reported structures are more or less embedded into the regional schistosity, which dips 55°-70° towards 010°-020°. Zones 1 and 2 (for which historical reserves have been previously estimated) transect the claims in the West Nordeau area for 600 m along strike and have been intersected to 725 m depth. As pointed out by Jean (1990), the “en echelon” pattern of the lenses raises questions regarding the geological and assay continuity along strike because “ore grade” intersections may appear to be randomly located within the wider deformation corridors. Nevertheless, it is reported that structures occupied by Zones 1 and 2, separated by 25 m of pyroclastic rocks in the eastern part of the Properties, merge into a single structure in the western part of the Properties.

The other mineralized zone, Zone 3 and North Zone are less well understood as they are poorly defined and have irregular continuity and inconsistent gold mineralization. Zone 3, located south of Zone 1 and Zone 2, and 30 m south of a graphitic marker horizon, has been defined along two traces of about 100 m each. Sporadic intersections show that the North Zone occupies a position some 30 m north of structures Zone 1 and Zone 2. Selected best intervals from recent drilling by Plato on the West Nordeau occurrence are included in Item 6 of this report.

7.2.7.3 East Nordeau

The most significant mineralization underlying the claims in the East Nordeau area is found in three separate gold structures (1, 2 and 3) related to the upper iron formation of the Trivio Group sedimentary rocks, which consist of interbedded mudstones, siltstones, greywackes and iron formations. The three sub-parallel structures consist of gold-bearing, sulphide-rich quartz veinlets and veins that generally follow the stratigraphy and the well-developed regional E-W schistosity, dipping 50° to 75° north, and have an average true thickness of less than 2.0 m.

The mineralization consists of 1-5% disseminated sulphides or semi-massive sulphide veinlets (pyrite, pyrrhotite, arsenopyrite and traces of chalcopyrite) in association with quartz, chlorite, garnet and gold. Gold is found as free grains in quartz or as inclusions in the sulphide minerals (Rocheleau et al., 1988). Common alteration of wall rocks includes amphibolitization, chloritization, silicification and biotitization.

Zone 1 extends for 450 m laterally, whereas Zone 2, to the south of Zone 1, generally continues east-



west for 220 m. Both extend to a ca. 200 m depth. They parallel each other for some 130 m and are stratigraphically less than 30 m apart. Zone 3, which is further east and possibly in a stratigraphic position similar to Zone 1, has been traced for about 240 m laterally and to 150 m depth.

A fourth mineralized structure, underlying the claims in the East Nordeau area, carries erratic, low-grade gold values and occurs in a shear zone that transects mafic volcanic rocks south of the iron formation. This zone contains 1-5% disseminated sulphides in carbonatized and chloritized rocks with well-developed garnets.

7.2.7.4 East Bateman

The East Bateman occurrence lies 6.2 km east of the historic Chimo mine and 2.5 km east-northeast of the East Nordeau occurrence. A 1990 drilling campaign on East Bateman delineated two gold mineralized lenses in the south-central part of the claim group. Both lenses are associated with graphitic shales that are intruded by smoky quartz veins containing 2-5% disseminated arsenopyrite and minor free gold, which occurs as thin inclusions and coatings on the sulphide grains. The two zones are parallel and 10 m apart stratigraphically. They can be traced for about 100 m laterally and to a depth of about 50 m, ranging from 1.2 to 3.9 m thick. See Table 6.19 for the best historical drill hole intercepts from the claims in the East Bateman area. The best intersection was 3.9 g/t Au over 5.05 m, from 66.25 to 71.30 m in hole BA-88-14 (Perron, 1988; Boulianne, 1990).

7.2.7.5 West Simon

The West Simon occurrence is located approximately 1 km west of the closed Chimo Minemine. It is considered the western extension of the Chimo mine horizons because it exhibits similar geological and mineralogical characteristics. Mineralized zones manifest as gold-bearing sulphide (arsenopyrite) lenses, associated with iron formation within the central sedimentary unit (named Zone B in the Insmill Zone), and as chlorite and carbonate altered, gold-bearing silica-rich lenses in shear zones within the Chimo lavas. Mineralized zones occur as lenses parallel to stratigraphic units and schistosity. Several en echelon style lenses are interpreted over a strike of approximately 1.2 km. Zone A has been tested to 385 m depth and was intersected in a cross-cut during the 1988 underground program (SNC Inc., 1990). Zone B is mineralized over 125 m strike length, dips north at 70°, plunges steeply to the west and has been intersected at 170 m depth (SNC Inc., 1990). Zone C is 50 m below Zone B, within the Chimo lavas. It has been tested to a depth of 170 m and was partly explored by drifting in 1988 (SNC Inc., 1990).



7.2.7.6 Blue Grass

The Blue Grass occurrence lies 2.5 km along strike, WNW of the Chimo Mine deposit. This occurrence comprises quartz veins in a 1.5 m wide shear zone over a strike of approximately 450 m, flanked to the north and the south by volcanic rocks. Greywacke, argillite and conglomerate belonging to the Trivio Group host the mineralization, which consists of arsenopyrite, pyrite and traces of chalcopyrite. Arsenopyrite occurs as fine to coarse disseminations associated with parallel carbonate-altered shear zones injected with quartz. Native gold is reported in drill core and a few trenches (Blanchet, 1983; Vincent, 2015).

7.2.7.7 McDonough

The McDonough occurrence lies 4.5 km NW of the Chimo Mine and was discovered during a 1936-37 surface prospecting program. Intermediate volcanic flows and volcanoclastic rocks host the mineralization, which manifests as a quartz-filled shear zone fracture extending along strike for some 60 m (Lee, 1963). The main fracture is mineralized over a maximum width of 0.61 m (Tolman, 1940). The quartz infill vein contains tourmaline, pyrite, chalcopyrite, sphalerite and ankerite. Visible gold has been recorded in the hanging wall of the shear zone (Tolman, 1940). The altered quartz-carbonate zone enclosing the mineralized zone, 7.6 m wide, slopes 70° towards the north. Historical exploration results suggest that at a depth of 122 m, the zone shallows somewhat to a 60° dip to the north and reaches 10.5 m in width (Lee, 1963).

The main host rock of the mineralization is a coarse tuff that contains relatively large, disseminated fragments of coarse-grained andesite. The rock is generally strongly feldspathic and has undergone intense alteration. Mineralization is associated with the main fracture intersecting the tuffs. A discontinuous, 4.5 m wide lenticular porphyry mass appears parallel to the fracture zone ca. 9.0 m to the south. Along the roof of this mineralized fracture, a fairly large number of fractures intersect it and generally have undergone a slight displacement and contain narrow quartz lenses with varying amounts of tourmaline. Both ends of the main fracture appear to divide or branch into similar subsidiary fractures. In addition, there are transverse veins of quartz only a few centimetres wide, typically filled with white quartz and barren of tourmaline. (Tolman, 1940).

7.2.7.8 South Gold Zone

This showing is located on a block of nine (9) claims owned by Texas T. Minerals Inc. (15%) and G.E.T.T. Or Inc. (85%), located within the eastern part of the Properties approximately 12 km ESE of



the Chimo Mine deposit. A mineralized E-W shear zone hosts two thin horizons of bedded and foliated sulphides separated by a zone of fractured rock. The mineralized horizons are composed of sulphides interbedded with quartz, chert and sericite, imparting a banded appearance. The lithology associated with mineralization, mainly identified by drilling, comprises fragments of various compositions and sizes in a micaceous matrix. The composition of the matrix varies from chlorite to chlorite-amphibole-garnet and finally to biotite-garnet, with or without sulphides (e.g., pyrite, pyrrhotite and chalcopyrite; Lafontaine and Tremblay, 1996). High gold grades are accompanied by significant copper mineralization (Tremblay, 1996). Pyrrhotite is in the form of layers or nodular masses. Pyrite is found as fine cubes, infillings along fracture planes, veins, folded layers near fault planes, and replacement crowns around pyrrhotite nodules. Chalcopyrite is observed within sulphide layers near shear zones and veins intersecting the host rock (Lafontaine and Tremblay, 1996). The mineralized shear zone is cut and displaced by NE and NW faults (Lafontaine and Tremblay, 1996). The Grenville Front is located a few kilometres southeast of the deposit.



8 DEPOSIT TYPES

The following description is taken from Langton et al. (2019) unless indicated otherwise.

Archean orogenic gold deposits are generally defined as structurally controlled vein or shear-margin deposits emplaced epigenetically in all lithologies occurring in Archean volcano-plutonic belts (Groves et al., 1998). These gold concentrations are the result of relatively homogeneous hydrothermal fluid flows of variable origin, including metamorphic devolatilization, felsic plutonism and mantle fluids (Hagemann and Cassidy, 2000).

Orogenic gold deposits are emplaced along active convergent margins during compressive tectonic regimes (Groves et al., 1998). This setting promotes hydrothermal fluid flow along major dislocation zones, which serve as structural traps for gold that precipitates out of solution. The importance of these structures is very clear in the Abitibi, where the vast majority of mines are located within 5 km of major structural discontinuities; however, relatively few deposits are situated at the heart of the main conduits (Eisenlohr et al., 1989; Groves et al., 1989; Robert, 1990), but are preferentially deposited along second- and third-order structures of the regional fracture/shear network, close to the large-scale compressive structures.

Structural control is predominant at both the mesoscopic and macroscopic scales of mineralization. The brittle to ductile nature of the structural controls is expressed in a wide variety of styles, including (a) brittle faults in ductile shear zones indicating low- to high-angle reverse movement, strike-slip or oblique movement; (b) networks of fractures, stockworks or brecciated zones in competent rocks; (c) foliated zones; and (d) fold hinges in ductile turbidite and iron formation sequences (Groves et al., 1998).

Orogenic gold deposits exhibit strong hydrothermal alteration with lateral zoning composed of mineral assemblages indicative of proximal to distal alteration. These assemblages, composed generally of carbonates (ankerite, dolomite or calcite) and sulphides (mainly pyrite, pyrrhotite and arsenopyrite), vary with the type of host rock and crustal depth. Alkaline metasomatism is characterized by sericitization or albitization or by the formation of fuchsite, biotite, alkaline feldspars and/or by chloritization of mafic minerals. Sulphidation reaches a peak in iron formations and iron-rich host rocks. Greenschist facies alteration of host rocks implies the addition of significant quantities of CO₂, S, K, H₂O, SiO₂, ±Na and light lithophile elements (Groves et al., 1998). The Project has geological potential for two main types of orogenic gold deposits: Type 1: greenstone-hosted quartz-carbonate vein type (Dubé and Gosselin, 2007); and Type 2: BIF-hosted



gold mineralization type (Robert et al., 2007).

8.1 Type 1: Greenstone-Hosted Quartz-Carbonate Vein Type

Type 1 gold deposits comprise structurally controlled gold mineralization in altered high-strain (“shear”) zones infilled with quartz or quartz-carbonate veins parallel with the shear zones, which are most likely to be within the volcanic units. Associated disseminated sulphides include arsenopyrite, pyrite and minor chalcopyrite. Graphitic horizons are common.

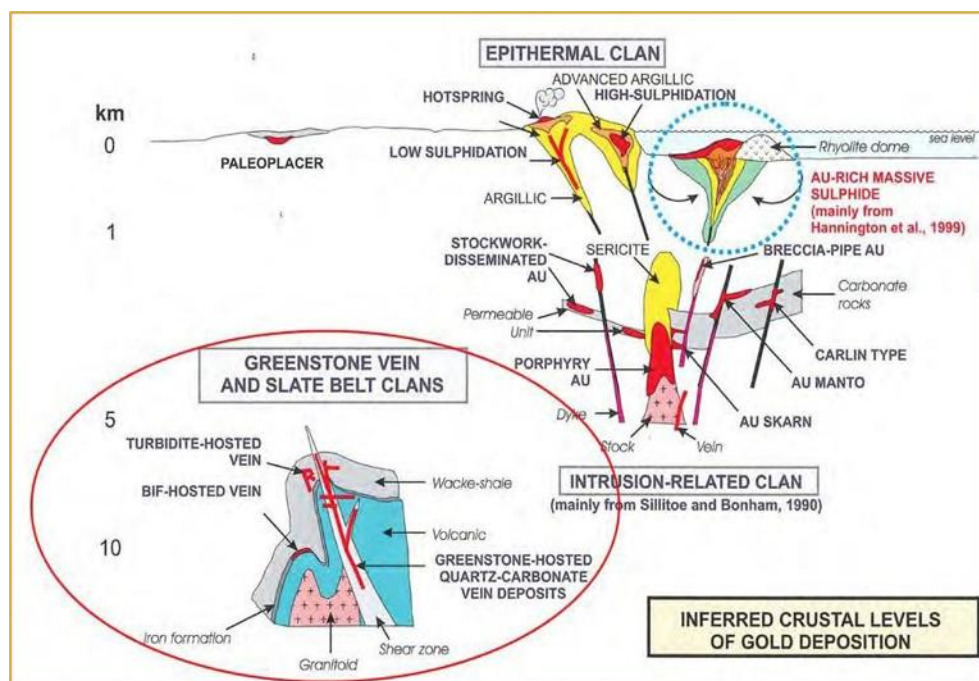
The following description is modified from Dubé and Gosselin (2007):

Greenstone-hosted quartz-carbonate vein deposits typically occur in deformed greenstone belts of all ages, especially those with variolitic tholeiitic basalts and ultramafic komatiitic flows that are intruded by intermediate to felsic porphyry intrusions, and sometimes with swarms of albitite or lamprophyre dykes. These types of deposit are distributed along major compressional to trans-tensional crustal-scale fault zones in deformed greenstone terrains, commonly marking the convergent margins between major lithological boundaries, such as volcano-plutonic and sedimentary domains. The large, greenstone-hosted quartz-carbonate vein deposits are commonly spatially associated with fluvio-alluvial conglomerate distributed along major crustal fault zones. This association suggests an empirical time and space relationship between large-scale deposits and regional unconformities.

The greenstone-hosted quartz-carbonate vein deposits are structurally controlled complex epigenetic deposits characterized by simple to complex networks of gold-bearing, laminated quartz-carbonate fault-fill veins. These veins are hosted by moderately- to steeply-dipping compressional brittle-ductile shear zones and faults with locally associated shallow-dipping extensional veins and hydrothermal breccias. The deposits are hosted by greenschist to locally amphibolite-facies metamorphic rocks of dominantly mafic composition and formed at intermediate depth (5-10 km). The two described Au-deposit types, which belong to the Greenstone Vein and Slate Belt “clans,” are shown in Figure 8-1 at their inferred crustal level of formation. Although sulphides are associated with the gold mineralization on the Properties, the discovery of significant base-metal deposits on the Properties is not likely, as its geological environment appears to be relatively distal to any paleo-volcanic center.



Figure 8-1 : Schematic illustration of the various types of gold deposits, shown at their inferred crustal levels of formation (modified from Poulsen et al. (2000) and Robert (2004))



8.2 Type 2: BIF-Hosted Gold Mineralization Type

Type 2 gold deposits are hosted in, or spatially associated with, banded iron formation (“BIF”). Gold mineralization is generally located in silicified lodes with disseminated to semi-massive sulphides (arsenopyrite, pyrrhotite and pyrite) spatially related to the BIF. Secondary quartz veining is commonly associated with this type of mineralization.

The following description is modified from Robert et al., 2007:

The deposits consist mainly of sulphidic replacements of Fe-rich layers in magnetite- or silicate-BIF, containing variably-developed quartz veins and veinlets. The intensely mineralized central parts of some deposits consist of nearly continuous wall rock replacements that can obscure their epigenetic character and can lead to ambiguities about the timing of mineralization.

BIF-hosted deposits occur in greenstone belts that are either volcanic-dominated or sediment-dominated, where they are located stratigraphically near regional volcanic-sedimentary transitions. These types of deposit may also occur near the edges of large clastic sedimentary basins, in the absence of significant mafic volcanic rocks. Magnetite-BIF is the dominant host in greenschist grade rocks, whereas silicate-BIF prevail in rocks of mid-amphibolite grade or higher (Kerswill, 1996).

At the local scale, BIF-hosted deposits are commonly associated with the hinge areas of folds, and





with intersections of shear zones and faults. As a consequence, the deposits are commonly stratabound and plunge parallel to their host fold hinge, or to the line of intersection of controlling shear zones with the BIF unit. In greenstone belts, many BIF-hosted deposits also contain concentrations of intermediate to felsic porphyry stocks and dykes.



9 EXPLORATION

This item documents the relevant exploration work conducted by the issuer. Item 6 (History) covers the exploration work conducted by previous operators on the Project.

The issuer's exploration activities have consisted almost exclusively of exploration and definition drilling (see Item 10) and drilling-related activities (drill rig access, collar surveying, down-hole surveys, etc.).

In February 2019, Cartier began compiling, interpreting and modelling the mineralized structures to build a 3D model using GeoticMine software.

Gold bearing structures were interpreted and updated based on structural, geological and gold-grade continuities. Important features used during the interpretation were:

- Interfaces of wacke/iron formation, wacke/basalt or iron formation/basalt;
- Graphitic marker horizon;
- Sulphides (mainly arsenopyrite);
- Veins and veinlets of smokey and/or milky quartz;
- Alteration (silica, biotite, chlorite and carbonate);
- Shearing and deformation interfaces.

Since 2019, Cartier's team has been continuously compiling, interpreting and modelling in 3D the different mineralized zones of the Cadillac Project. This compilation and exploration target generation work continues to this day and is constantly being updated.



10 DRILLING

This chapter presents the drilling program carried out by Cartier between 2016 and 2024 on the Cadillac Project.

Refer to Chapter 6 (History) for the drilling programs from previous operators.

10.1 Drilling Methodology

Cartier uses the Devisoft software and the Geotic software suite (GeoticLog, GeoticGraph, GeoticCAD and GeoticMine) for all geomatics operations.

Drilling programs were performed by Forage Orbit Garant, with the collection of HQ, NQ et BQ size core.

A network of forestry trails and roads provides access to some drill sites. These have been restored to a usable condition to minimize the environmental impact and maximize employee safety. New access roads were built to reach other sites. Any trees, shrubs and alders growing on the drill sites or access roads are shredded by contractor F. Alarie of Val-d'Or and used as ground cover.

At the end of each drilling program, Cartier closes the drill sites by inspecting the area and removing any waste left behind after the rig is demobilized. Anchor casings are left in place and secured with bolted steel caps to prevent debris from falling inside, except for abandoned holes. Aluminum tags with the engraved hole number are attached to the base of the casing and the top of the 2-m rod connected to the steel cap. Drill sites with suitable soil composition and light exposure are seeded.

10.1.1 Drillhole Location/Set-up

Drill collars are positioned using three comparable readings taken with a Garmin 60CSx GPSmap (coordinate system: UTM, NAD 83, Zone 18). The collars are then marked with a wooden stake flagged with fluorescent orange tape inscribed with the hole number and the intended direction and plunge of the hole. A TN14 Gyrocompass from Reflex Instruments is used to align the drill rig. The gyroscope in this device detects geographical north by its sensitivity to the Earth's rotation. It is not affected by interference from highly magnetic ground or the drift effect.



10.1.2 Drillhole Orientation during Operation

The first deviation measurement in each hole is taken with an EZ-GYRO device (Reflex Instruments) 9 m past the bedrock contact. Drilling continues if the value corresponds to the desired azimuth and plunge. If the value is too far off (azimuth and/or plunge), the hole is restarted until the measurement in the bedrock is satisfactory.

Deviation tests are then carried out every 3 m, 6 m or 9 m down the hole, depending on whether one or two core barrels are used.

Despite these protocols, some holes still deviate from the intended trajectory. In such cases, Cartier uses Devico's DeviDrill technology to correct the hole plunge or azimuth and quickly reposition the hole along the planned trajectory. The DeviDrill tests are carried out every 3 m to quickly obtain deviation readings and determine the next action to take.

At the start of each day or when a DeviDrill intervention is underway, the project geologist collects the readings from the drilling foreman. Once filtered and validated, the deviation data are added to the GeoticLog database. Cartier's senior geomatics geologist can generate the drill hole trace and, if necessary, stop the hole to reposition it.

10.1.3 Drilling and Core Handling

The core is collected in a standard drilling tube and the driller's helper carefully places the core into wooden core boxes at the drill rig. The helper also marks the depth (m) after each 3-m run with wooden blocks and closes the box with fiber tape. Core trays are numbered with a permanent marker indicating the drillhole number and the sequential box number.

Generally, the drillhole is stopped at a specific depth determined by the project manager or following the instructions of the field geologist.

Once the drillhole is completed and the final downhole survey reading is collected, the drill crew pulls the rods for mobilization to the next drill site. A metallic cap with a metal tag displaying the hole number is put on the collar of the hole. All casing has been left in place, except for the drillholes that had been stopped and restarted due to a bad orientation or dip. No drillholes have been grouted or cemented.

10.1.4 Core Logging and Measurement

A detailed log of the drill core is documented by experienced and qualified geologists who are members in good standing of the OGQ. Geologists record their descriptions of lithological units,



alteration, structures, veins and mineralization in GeoticLog software.

The core boxes, are displayed on tables in rows of four or five for core logging. Geologists check the box numbers and the markings on the blocks inserted by the driller helper for any errors in numbering or footage. The core is aligned, and the pieces are fitted together to eliminate gaps. The footage interval of each box is recorded in the log. Lastly, the core is wetted, and a single photograph of each row of boxes is taken.

10.1.4.1 RQD and Core Recovery

RQD and core recovery are estimated for mineralized zones and their wall rocks (over a 15-m core length on each side of the mineralized zone). RQD is calculated by measuring each section of core 10 cm or longer. These sections are summed within each interval of 3 m, the distance between two blocks of wood (i.e., a drilled interval) and represented as a percentage. Core recovery is also calculated as a percentage. Recovery of 100% means that 3 m of core has been placed into the box between the two blocks of wood, representing a 3-m run.

10.1.5 Core Photography

Once logged by the geologist, all drill core is photographed. The objective of core photos is to have a digital image recorded with sufficient details to clearly see core features prior to destructive sampling procedures. This record can be used later to qualify rock quality features and to examine core images against geological logging if the core is unavailable for examination. The photos are also used, as required, during the creation of geological sections.

Once the core is photographed, the boxes are placed on a mobile rack until it is assigned to the core saw operator for splitting and sampling on site.

10.1.6 Core Storage

Technician attaches a Dymo-embossed aluminum tag to the front of each box containing any core of interest (mineralization and/or characteristic stratigraphy typical of the sector). The remaining core boxes are properly disposed of. The aluminum tag displays the drill hole number, box number and from-to interval. After each drilling program, all boxes of barren core (no significant gold values) or core of no interest to the current drilling objective are placed on securely wrapped and tied wooden pallets and temporarily stored outdoors on the premises of MNG Services Ltd ("MNG") in Val-d'Or.



Boxes containing the core of interest (numbered boxes with aluminum tags) are stacked on pallets, wrapped, tied and placed in medium- to long-term storage inside the MNG facility. A Cartier employee writes (with a permanent black marker) the drill hole ID and/or the pallet number on the pallet to quickly track it down if needed for review. The best gold-bearing sections are kept inside Cartier's core shack for quick and easy access as needed.

10.2 2016-2024 Drilling Programs

Cartier initiated its first drilling program on November 1, 2016. Drilling was divided into five phases on the Cadillac property (Table 10-1). The objective was to expand the known gold zones and enhance the discovery potential for new gold zones.

Table 10-1: Summary of the drilling completed by Cartier since 2016

Phase	Year	No. of holes	Total length (m)	No. of samples for gold analysis (excl. QA/QC)
1	2016-18	72	34,332	13,776
2	2018-19	33	13,248	4,502
3a	2019	4	1,663	707
3b	2019-20	15	8,810	2,880
4	2024	162	27,927	7,103
Total		286	85,980	28,968

Phase 1, completed by August 2018, consisted of 72 holes totalling 34,332 m. The first objective was to test the geometric extensions of Zones 5B, 5B2, 5C, 5M, 5M2 and 5N (Structures 5B, 5B2, 5C, 5M, 5M2 and 5N of the Central Corridor) below the old Chimo mine between depths of 900 and 1,500 m. Holes CH17-46 and CH17-47 account for a third (10,113 m) of Phase 1 drilling. The second objective was to delineate the geometric extensions of the satellites Zones 2, 2B, 2W, 3, 3W, 3E, 4B and 4B2 between depths of 200 and 700 m (Structures 2, 3 and 4B of the North Corridor), Zones 5B3, 5B4, 5M3, 5M4, 5NE and 6N1 between depths of 200 and 500 m (structures 5B, 5M, 5N and 6N1 of the Central Corridor) and zones 6, 6B, 6P and 6P2 between depths of 300 and 600 m (Structures 6, 6B, 6P and 6P2 of the South Corridor).

Phase 2, which took place between July 2018 and February 2019, consisted of 33 holes totalling 13,248 m. The objective was to expand the geometry of Zones 2B and 3E between depths of 400 and 600 m (Structures 2 and 3 of the North Corridor), Zones 5B4, 5M4, 5NE and 6N1 between depths of



600 and 1,100 m (Structures 5B, 5M, 5N and 6N1 of the Central Corridor), and Zone 6P2 between depths of 300 and 700 m (Structure 6P2 of the South Corridor).

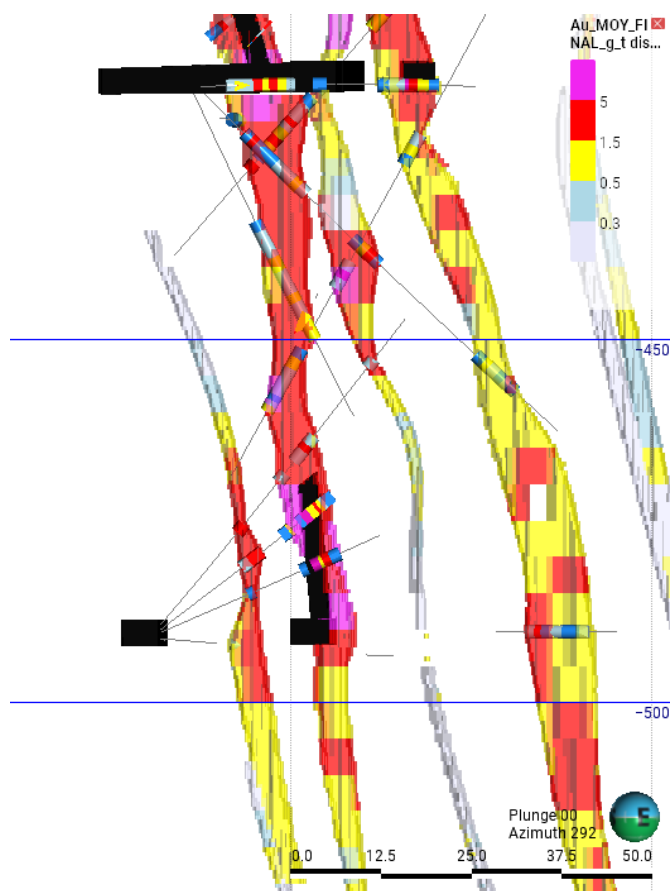
Phase 3a was conducted from February to May 2019, and consisted of four holes totalling 1,663 m. The objective was to test, in the eastern part of the Chimo Mine property, the geometric extensions of Zones 5B4, 5M4 and 5NE between depths of 600 to 800 m (Structures 5B, 5M and 5N of the Central Corridor). At the same time, the holes crossed and tested the North Corridor.

Phase 3b, conducted from November 19, 2019, to June 27, 2020, consisted of 15 holes totalling 8,810 m. The holes were drilled in the eastern part of the Chimo Mine property and demonstrate the continuity of mineralization in zones 5B4, 5M4 and 5NE over a depth of 1.3 km. Drilling also led to the discovery of Zone 5CE and revealed the potential to add resources to this part of the property.

Phase 4, conducted from February to December 2024, consisted of 162 holes totalling 27,927 m. The main objectives of the drilling campaign were to increase the potential for expanding the size of known gold zones and to highlight the potential for discovering new gold zones. Drilling validated the project's geological model and helped increase the size of known gold zones and discover new zones with high gold content. Figure 10-1 shows a typical cross section.



Figure 10-1: Cross section looking NW with drillholes, assay, historical voids (black) and interpolated blocks



11 SAMPLE PREPARATION, ANALYSES AND SECURITY

The following sections describe Cartier's core handling, sampling, and security procedures for the most recent completed diamond drilling program that took place in 2024.

Drill programs from 2016 to 2020, including the QA/QC procedures and results, and those of Plato Gold Corp. from 2007 to 2009, Chalice Gold Mines Ltd ("Chalice") from 2017 to 2019 and O3 Mining Inc. ("O3 Mining") in 2020 were covered in the previous Technical Report on the Project.

11.1 Core Handling, Sampling, and Security

The core samples are placed in boxes and sealed at the drill rig, then transported daily to the logging facility in Val-d'Or, where a technician takes over to handle them. The core samples are logged and sampled by professional geologists or, under their direct supervision, by trainee geologists. After logging, the cores are marked with red grease pencil for analysis. As a general rule, only mineralized zones are sampled. Sampling intervals follow lithological and/or alteration contacts in order to be as representative as possible.

The intervals for each sample are recorded in the drilling log, using GéotricLog software, as well as in the sample logbook provided by Imprimerie Harricana in Amos. The logbook has three sections. The sampled interval, as well as the project name, drill hole number, date, and type of analysis required, are recorded in the first section. Only the sampled interval and the type of analysis are mentioned in the second section. Finally, the third section of the notebook is left blank, except for the type of analysis required. The first section is left in the notebook, while the other two are detached from the notebook and placed in the core box at the beginning of each sample. When sawing the sample, the second section is stapled into the box and acts as a reference or control, while the third section is placed in the sample bag to be sent to the analysis laboratory.

Finally, the core is marked with a blue grease pencil for lithogeochemical sampling (major and trace element content). The length of a sample is always the same, i.e., 0.2 m. The rest of the procedure is exactly the same as for metal sampling.

The possibility of contamination or error in the analytical process means that Cartier closely monitors the analysis results sent to it by the laboratory. As a result, Cartier includes sterile (blank) samples and certified reference material (standards) in its shipments. In summary, for a shipment of one hundred (100) samples, no fewer than five (5) blanks and five (5) standards were automatically



included with the core samples.

For verification purposes, Cartier uses blanks consisting of marble samples purchased from hardware stores, which are assumed to be gold-free, as well as various standards provided by Analytical Solutions Ltd. (ASL), a laboratory specializing in this field, whose gold content is known with great precision. The latter come in pre-prepared bags, sealed by ASL and weighing 60 g each.

11.1.1 Gold Assays Samples

Once the samples have been marked by the geologist in charge, the core boxes are transported to the core sawing room. Sawing is carried out by qualified technicians or laborers, and safety checks are monitored by the geologists responsible for describing the core.

The core is then sawn lengthwise along a red line that has been marked in advance by the geologists. The technician takes care not to move the sample tag placed by the geologists at the beginning of each sample. Once the core has been completely sawn, one half of the core is inserted into a transparent plastic bag containing the third section of the sample tag (the one on which only the type of analysis required was written) and on which the technician has previously written the sample number with a permanent marker. The sample number corresponds to the one written on the sample tag by the geologists. The other half of the sawn core is left in the box and can be used as a control if a review of the core proves necessary. The second section of the tag is stapled to the bottom of the box, marking the beginning of each sample.

The technician then seals the plastic bag with staples. Once seven samples have been sawn, they are placed in a polypropylene bag on which the company name, sample intervals, and number of samples are written with a permanent black marker.

A shipping form is filled out by the project geologist with information on the sample numbers, the number of samples, and the type of analysis required.

The samples are delivered to the Techni-Lab (Actlabs) laboratory in Val-d'Or for preparation, and the pulp obtained is shipped to their laboratory in Ste-Germaine-Boulé for analysis.

When the samples are received at the laboratory, a laboratory worker verifies the conformity of the shipment and sends a LIMS (Laboratory Information Management System) file to Cartier's project geologist, where the analytical requests and requirements are recorded and confirmed.

Once the analysis results have been obtained and the final certificate signed by the chemist has been sent, all pulp returned by the laboratories is automatically disposed of in a container. Analysis rejects are also discarded, except for gold-bearing mineralized zones of interest. The retained rejects



are placed on wooden pallets, numbered with permanent black marker, wrapped, tied, and stored indoors for the medium to long term at MNG storage facility in Val-d'Or.

11.2 Laboratories Accreditation and Certification

The 2024 drill program samples were shipped to Techni-Lab (now Actlabs) in Val-d'Or. The Techni-Lab (Actlabs) laboratory is ISO/IEC 17025:2005 accredited for geochemical analysis. This certification requires proof of quality control covering all aspects of the organization. It is an independent commercial laboratory.

11.3 Laboratory Preparation and Assays

All samples received are processed through a sample tracking system, which is an integral part of rigorous quality control monitoring (LIMS). This system uses graphic barcodes to identify each sample received. This allows a given sample to be tracked throughout the preparation and analysis process, which also prevents errors when sample numbers are entered manually into an Excel file.

11.3.1 Sample Analysis Procedure

The samples received are first dried, then ground to a degree where more than 80% of the sample has a particle size less than or equal to -8 mesh (0.07 mm). Part of this sample, i.e., 500 g, is then pulverized until 90% has a particle size less than or equal to -200 mesh (0.07 mm). Only 50 g of this 500 g will be used for the actual analysis (code RX-1 (500)). The remaining 450 g are returned as pulp to the Cartier office, along with the original sample reject.

The equipment used to grind and pulverize the sample is thoroughly cleaned with a sterile cleaner after each batch of 24 samples or after each highly mineralized sample. The sample preparation stations are also equipped with dust recovery systems, reducing dust levels and the risk of contamination.

A batch of 24 samples includes 20 Cartier samples and four Techni-Lab reference materials (blank, standard, and duplicate) used for quality control. If a reference material has an erroneous value or exceeds the accepted control limits, an error report is immediately generated. This allows the person responsible for evaluating the analyses to immediately identify a problem related to a specific batch of analyses and correct the situation.

Gold analysis (metal sampling; code 1A2-50) is performed by melting a 50 g sample by pyroanalysis



and reading the concentrations by atomic absorption (AA). Samples with results between 1.0 and 5.0 g/t gold were reanalyzed by atomic absorption (code 1A2-50), and those with values above 5.0 g/t gold were reanalyzed by gravimetry (code 1A3-50). For samples containing visible gold, 500 g of rock is analyzed using the “Metallic Sieve” method (code 1A4).

11.4 Quality Assurance and Quality Control

NI 43-101 requires mining companies reporting results in Canada to follow CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (“CIM Best Practice Guidelines”). The guidelines describe which items are required to be in the reports but do not provide guidance for quality assurance and quality control (“QA/QC”) programs.

QA/QC programs have two components. Quality assurance deals with preventing problems using established procedures, while quality control aims to detect and assess problems and take corrective actions. QA/QC programs are implemented, overseen and reported on by a QP as defined by NI 43-101.

QA programs should be rigorous, applied to all types and stages of data acquisition, and include written protocols for sample location, logging and core handling, sampling procedures, laboratories and analysis, and data management and reporting.

QC programs are designed to assess the quality of analytical results for accuracy, precision and bias. This is accomplished through the regular submission of standards, blanks and duplicates with regular batches of samples submitted to the laboratory and the submission of batches of samples to a second laboratory for check assays.

The materials conventionally used in mineral exploration QC programs include standards, blanks, duplicates, and check assays. The definitions of these materials are presented below:

- Standards are samples of known composition inserted into sample batches to independently test the accuracy of an analytical procedure. They are acquired from a known and trusted commercial source. Standards are selected to fit the grade distribution identified in the mineralization.
- Blanks consist of material that is predetermined to be free of elements of economic interest to monitor for potential sample contamination during analytical procedures at the laboratory.
- Duplicate samples are submitted to assess assay precision (repeatability) and



mineralization homogeneity. Duplicates can be submitted from all stages of sample preparation with the expectation that better precision is demonstrated by duplicates further along in the preparation process.

The QA/QC program implemented by Cartier includes the insertion of blanks and standards in its sampling protocol. During the drilling work carried out by Cartier, a total of 11% of all samples collected were used for QA/QC (Table 11-1).

Table 11-1: Summary of QA/QC samples collected during the 2024 drilling program

Sample Count	QAQC Sample Count		QAQC Sample %	
	Blank	Standard	Blank	Standard
7 103	394	397	5%	6%
Total	791		11%	

11.4.1 Blanks

Blanks are used to monitor for potential sample contamination that may take place during sample preparation and/or assaying procedures at the laboratory. Samples of barren crushed white quartz or carbonate (blank) were used.

In the sampling sequence, one blank sample is inserted every 20 samples.

Each time a certificate of analysis is received, it is checked in detail. The tolerance is three times the detection limit for blanks (i.e., 24 ppb Au for Techni-Lab). Beyond this tolerance, the entire batch from which the invalidated sample comes is reanalyzed. However, if a high value precedes or follows it, a greater tolerance is allowed and does not necessarily require reanalysis.

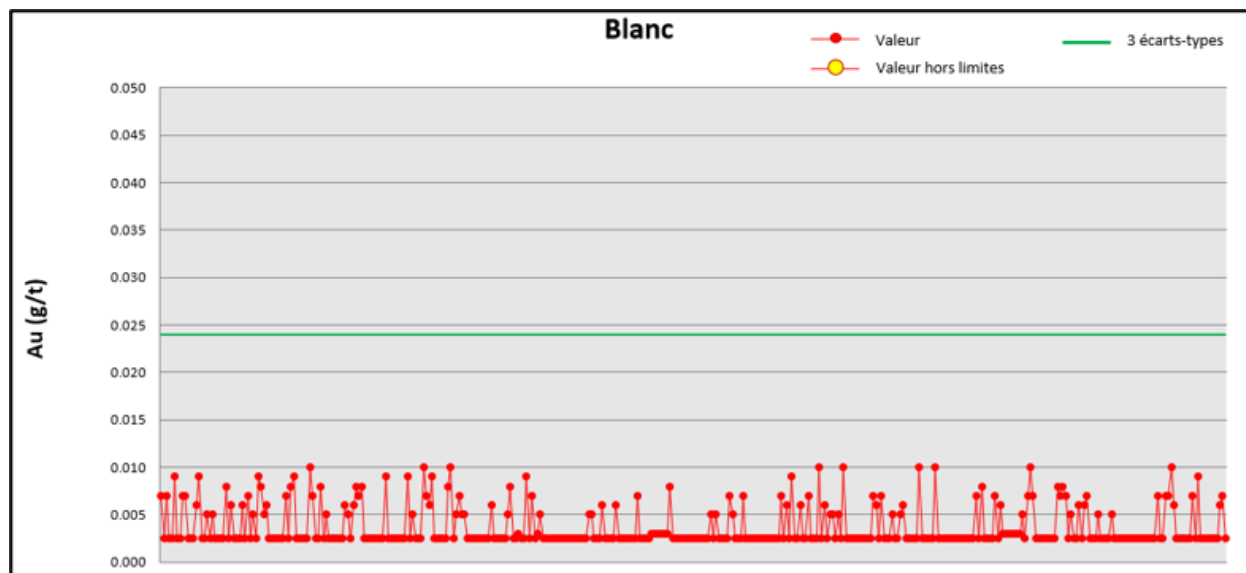
During drilling, a total of 394 blank samples were collected (Table 11-2). No samples showed contamination (> 24 ppb Au; Figure 11-1).

Table 11-2: Results of blanks from the 2024 drilling program

Blanks Count	Valid Count	Invalid Count	Valid %
394	394	0	100%



Figure 11-1 : Results for blanks used during the 2024 drilling program



11.4.2 Standards

Accuracy and precision are monitored by the insertion of CRMs. A suite of commercially available CRMs was used (Table 11-3).

In the sampling sequence, one standard sample is inserted every 20 samples.

Each time a certificate of analysis is received, it is checked in detail. The tolerance is three standard deviations for standards. Beyond this tolerance, the entire batch from which the invalidated sample originates is reanalyzed. If a sample preceding or following the failed standard does not return an abnormal gold value, it is not necessarily retested.

During the 2024 drill program, six different standards were used and inserted into the sampling process (Table 11-3).

Table 11-3: List of standards used during drilling in 2024

Standard	Certified Grade (g/t)	Average Grade Techni-Lab (g/t)	Standard Deviation (g/t)	- 3 Standard Deviation (g/t)	+ 3 standard Deviation (g/t)
608	1.210	1.078	0.040	1.090	1.330
236	1.850	1.817	0.059	1.673	2.027
240	5.510	5.426	0.139	5.093	5.927
240B	5.648	5.615	0.143	5.219	6.077
243	12.390	12.259	0.306	11.472	13.308
611	15.700	14.708	0.600	13.900	17.500

A total of 397 standards were collected (Table 11-2). A total of 45 samples show a slight analytical accuracy issue (Figure 11-2 to 11-7). Since these problematic values were not located in gold-



bearing areas and there were no anomalous grades located before or after, no assay were re-taken.

Figure 11-2: Validation graph for standard 608

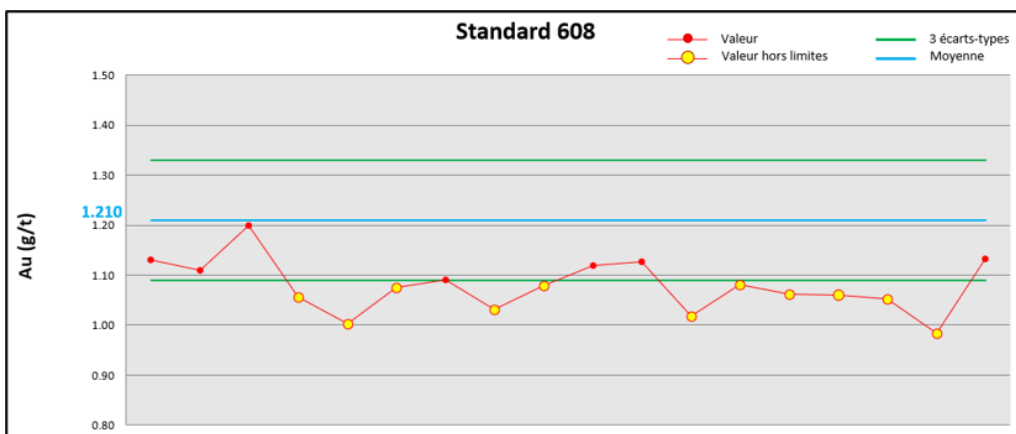


Figure 11-3: Validation graph for standard 236

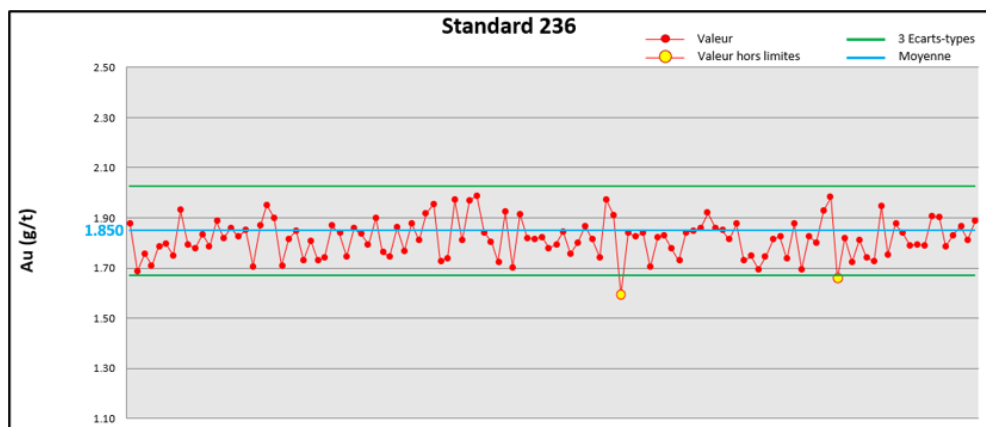


Figure 11-4: Validation graph for standard 240

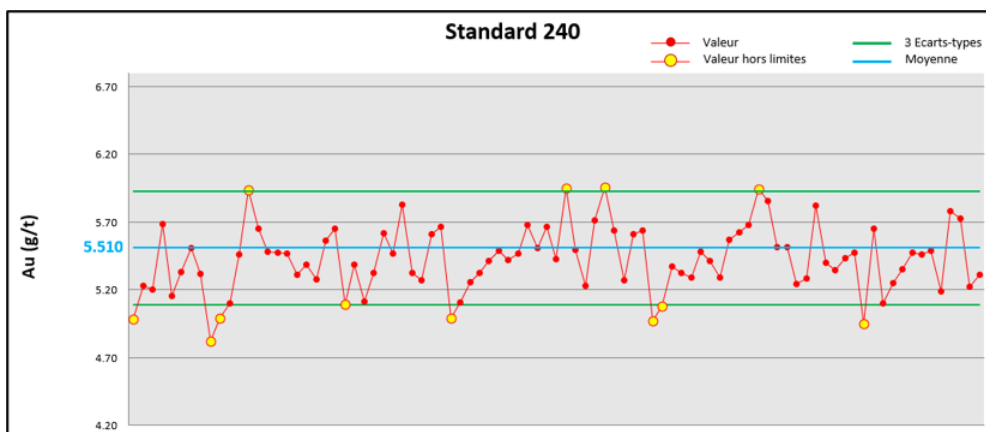


Figure 11-5 : Validation graph for standard 240B

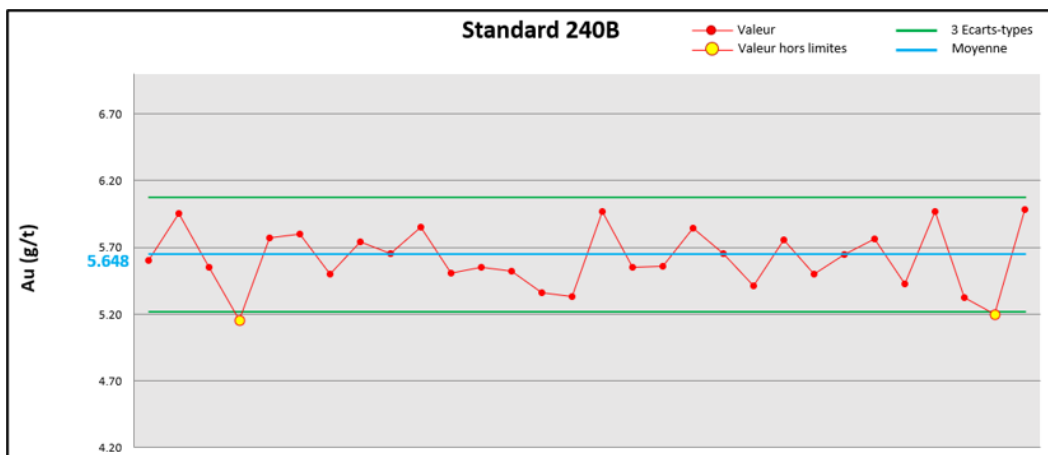


Figure 11-6: Validation graph for standard 243

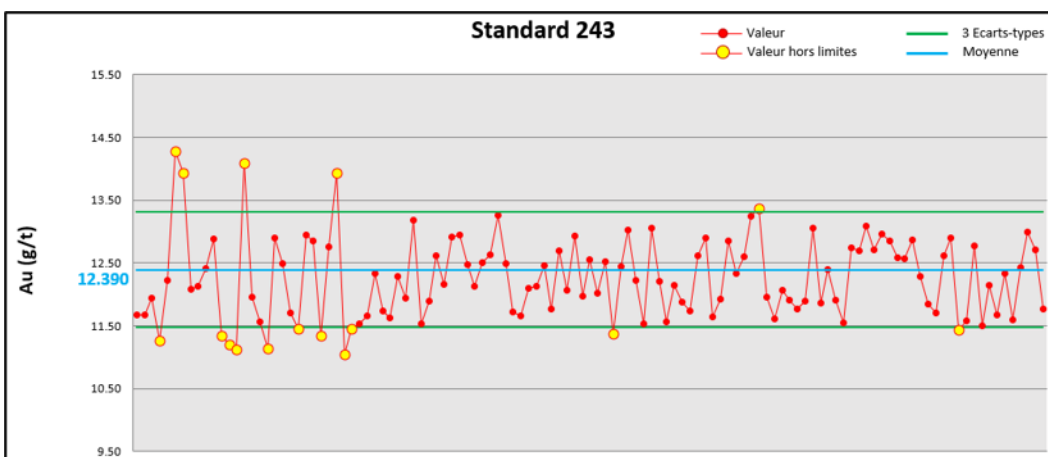
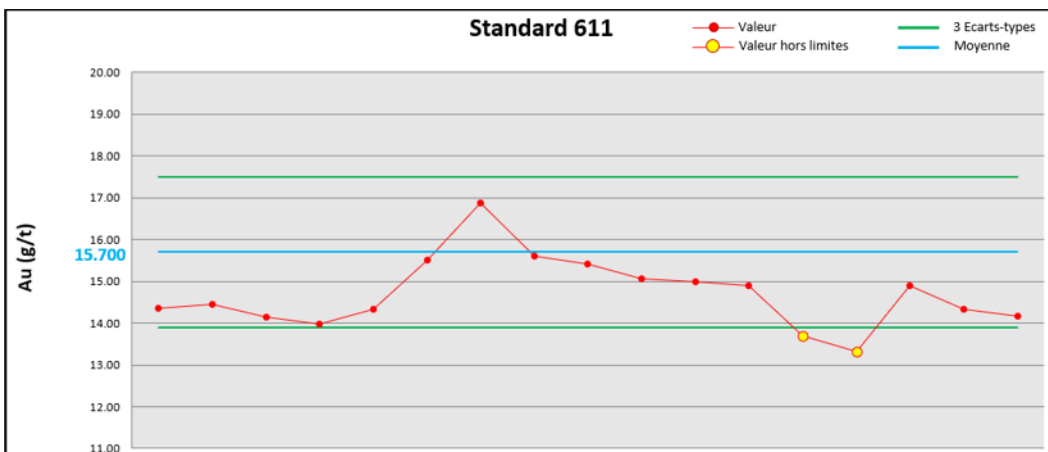


Figure 11-7: Validation graph for standard 611



11.5 Conclusion

The QP reviewed the sample preparation, analytical and security procedures, as well as insertion rates and the performance of blanks, standards, and duplicates for the 2024 drilling program. The QP concluded that the observed failure rates are within expected ranges and that no significant assay biases are present.

The QA/QC data indicate that the overall assay results of the issuer's drill program are valid and can be relied upon for the purpose of this Report.

It is the QP's opinion that the sample preparation, security and analytical procedures are adequate and follow best practices. However, the QP recommends adding core duplicate samples and check assays in a second laboratory to the QAQC procedures.



12 DATA VALIDATION

For the purpose of this Technical Report, PLR performed a basic verification on the entire Project database. All data was provided by Cartier in UTM coordinates. The database close-out date for the resource estimate is February 17, 2025; data from 4,477 diamond drillholes (DDH) (535,360 m) and 167,978 assays is part of the database.

12.1 Site Visit

Pierre-Luc Richard of PLR visited the Project in May 2025, during the course of this mandate. The site visit included a visual inspection of historical core drilled by past owners and recent core drilled by the issuer, as well as a field tour and discussions of geological interpretations with the issuer's geologists and management. The site visit also included a review of sampling and assaying procedures, the quality assurance / quality control ("QA/QC") program, downhole survey methodologies, and the descriptions (logging) of lithologies, alteration and structures. Selected drill collars were also validated in the field using a handheld GPS.

12.2 Sample Preparation, Analytical, QA/QC and Security Procedures

Cartier procedures are described in Chapters 10 and 11 of the current Report. Discussions held with on-site geologists confirmed that the procedures were adequately applied.

The QP Pierre-Luc Richard reviewed sections of mineralized core while visiting the Project. All mineralized core boxes were labelled and properly stored. Sample tags were present in the boxes, and it was possible to validate sample numbers and confirm the presence of mineralization in witness half-core samples from the mineralized zones (Figure 12-1).

The sample preparation room visited was clean and no issues were noted (Figure 12-2).

Analytical, QA/QC or security procedures are conducted in accordance with the industry standards.



Figure 12-1 : Mineralized core review during the site visit.



Figure 12-2 : Mineralized core review during the site visit.



12.2.1 Drillhole Location

Drill collars were positioned using three comparable readings taken with a Garmin 60CSx GPSmap (coordinate system: UTM, NAD 83, Zone 18). A TN14 Gyrocompass from Reflex Instruments is used to align the drill rig. The gyroscope in this device detects geographical north by its sensitivity to the Earth's rotation. It is not affected by interference from highly magnetic ground or the drift effect. Multiple casings were surveyed by the QP during the site visit and not discrepancies were noted.

Figure 12-3 : Collar location review during the site visit.



12.2.2 Downhole Survey

Deviation tests were carried out every 3 m, 6 m or 9 m down the hole using a EZ-GYRO device (Reflex Instruments). Some holes deviated from the intended trajectory and in such cases, Cartier used Devico's DeviDrill technology to correct the hole plunge or azimuth and quickly reposition the hole along the planned trajectory. The DeviDrill tests were carried out every 3 m to quickly obtain deviation readings and determine the next action to take.

Downhole survey data for the drilling programs were checked for discrepancies. Spurious measurements were tagged by Cartier's geologists as "false" in the database and were not considered by the software for the modelling.

12.2.3 Assays

PLR was granted access to the original assay certificates directly from laboratories for all the holes requested on the Project. All certificates received were verified against the database. No major discrepancies were noted. Values lower than the detection limits were set to half detection limit. In the assay table, the final gold value is calculated using a conditional priority. Metallic screen results have priority over the gravimetric finish results. The gravimetric finish results have priority over atomic absorption finish (AA). If more than one assay is done using the same analytical method, the average of the results is used but still considering the priority listed above.



12.2.4 Storage

All core boxes of deemed worth keeping are tagged and stacked on wrapped pallets, secured, and stored for the medium to long term inside MNG's premises in Val-d'Or. Cartier has marked these stored pallets with a black permanent marker, indicating the name of the drill hole and/or the pallet number so that the pallet can be quickly traced for review. The best gold-bearing mineralized zones are stored inside Cartier's core library for quick and easy access if needed (investor visits, company evaluations, other work). Rejects from mineralized zones are also placed on wooden pallets, numbered with black permanent marker, wrapped, tied, and stored indoors for the medium to long term at MNG's premises.

12.3 Conclusion

The QP is of the opinion that the drilling protocols in place are adequate. The database for the Cartier Project is of good overall quality. No material issues were noted during the validation process. In the QP's opinion, the Cartier database is appropriate to be used for the estimation of Mineral Resources.



13 MINERAL PROCESSING AND METALLURGICAL TESTING

As part of the Preliminary Economic Assessment, a metallurgical testwork program was completed using samples representative of the deposits included in the study—Chimo, East Nordeau, and West Nordeau. The program focused on whole-rock cyanide leaching and included gravity separation characterization, along with tests evaluating the effects of pre-aeration, lead nitrate addition, and elevated pH on gold extraction. Grindability was assessed using the Bond Ball Mill Work Index and SMC (SAG Mill Comminution) tests. Selected environmental characteristics of the cyanide leach tailings were also evaluated.

13.1 Previous Metallurgical Testwork

The Cadillac Project includes resources from the former Chimo Mine, which operated from 1989 to 1997 and had an on-site concentrator. Historical operating performance is not considered suitable for predicting future results because the mill used gravity concentration and flotation, followed by gold extraction from the concentrate at another mill owned by the operator. Metallurgical testwork was also conducted at SGS Lakefield before operations began; however, the results are not considered in the present study.

Pre-concentration testwork using mineral sorting technology was also conducted. However, because mineral sorting is not considered in the gold extraction assumptions used for the resource estimate, the results are not presented.

13.2 Testwork

A metallurgical testwork program was initiated in 2025 using resources scheduled for extraction over the life-of-mine plan. Gold from these resources is expected to be recovered by milling in an on-site process plant. The program results provide the basis for the gold recovery and other metallurgical performance assumptions used in the economic evaluation.

13.2.1 Samples Origin

The samples used in the metallurgical testwork were prepared from drill core collected from the three deposits considered in this study: Chimo, East Chimo, and West Nordeau. For each deposit, selected core intervals were combined to prepare two variability composites spanning a range of



gold grades. A master composite was then prepared for each deposit by blending equal proportions of the two corresponding variability composites, resulting in a total of six variability composites and three master composites.

The intervals were selected to represent the principal gold zones intersected within each deposit and to span the range of gold grades expected in the mill feed. Table 13-1: 2026 testwork program samples origin lists the drill core intervals used, their corresponding gold zones, and their respective lengths.

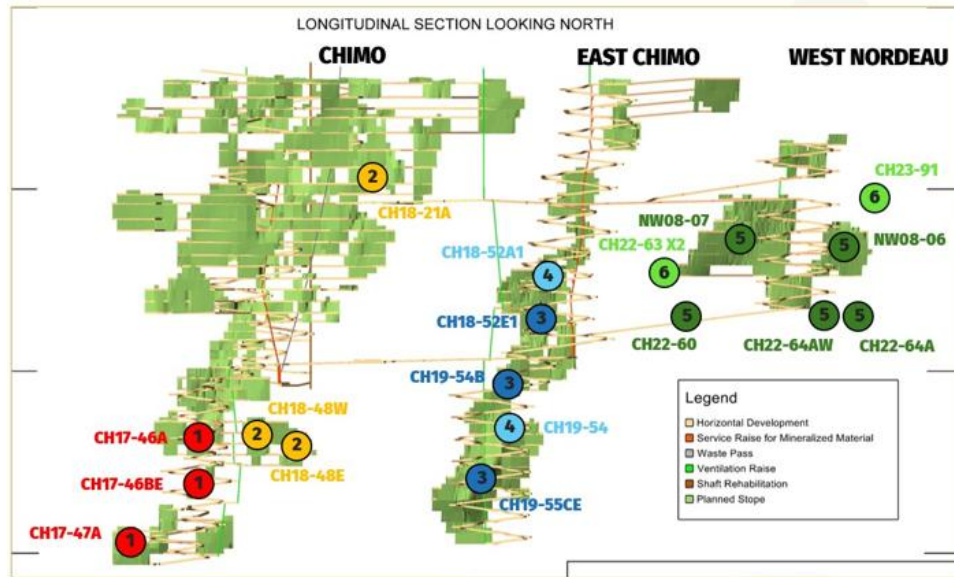
Table 13-1: 2026 testwork program samples origin

Composite	Hole ID	Gold Zone	From (m)	To (m)	Length (m)
CHIMO - 1	CH17-47A	5M2	1462.5	1469.5	7.0
CHIMO - 1	CH17-46A	5B	1152.0	1158.1	6.1
CHIMO - 1	CH17-46BE	5B	1250.0	1267.3	17.3
CHIMO - 2	CH18-21A	6N1	424.1	436.5	12.4
CHIMO - 2	CH18-48W	6N1	1121.0	1134.0	13.0
CHIMO - 2	CH18-48E	6N1	1174.0	1180.0	6.0
EAST CHIMO - 1	CH19-55CE	5B4	1321.0	1335.6	14.6
EAST CHIMO - 1	CH19-54B	5B4	1123.3	1135.1	11.6
EAST CHIMO - 1	CH18-52E1	5NE	768.0	773.5	5.5
EAST CHIMO - 2	CH19-54	5B4	1198.0	1214.1	16.1
EAST CHIMO - 2	CH18-52A1	5NE	711.0	724.0	13.0
WEST NORDEAU - 1	CH22-60	5NE1	834.4	839.0	4.6
WEST NORDEAU - 1	CH22-64AW	5NE2	736.0	742.0	6.0
WEST NORDEAU - 1	CH22-64A	5NE2	745.9	751.4	5.5
WEST NORDEAU - 1	NW08-06	5NE2	553.8	562.3	8.5
WEST NORDEAU - 1	NW08-07	5NE1	567.0	575.1	8.1
WEST NORDEAU - 2	CH23-91	5NE2	493.0	509.0	16.0
WEST NORDEAU - 2	CH22-63	4BE	554.0	561.0	7.0
WEST NORDEAU - 2	CH22-63	5NE1	614.0	619.6	5.6

Figure 13-1 shows the location of the drill core intervals (longitudinal view of the resources) and identifies the drill holes used to generate the composites.



Figure 13-1 : Location of the drill core intervals



13.2.2 Material Description

The drill cores were used to create two variability composites and one main composite for each zone. Table 13-2: 2026 testwork program samples composition summarizes the components of each composite.

Table 13-2: 2026 testwork program samples composition

Composite	Hole ID	Length (m)	Weight (kg)	Overall Weight (kg)
CHIMO - 1	CH17-47A	7.0	18.47	80.20
	CH17-46A	6.1	16.09	
	CH17-46BE	17.3	45.64	
CHIMO - 2	CH18-21A	12.4	31.67	80.20
	CH18-48W	13.0	33.20	
	CH18-48E	6.0	15.32	
EAST CHIMO - 1	CH19-55CE	14.6	36.94	80.20
	CH19-54B	11.6	29.35	
	CH18-52E1	5.5	13.91	
EAST CHIMO - 2	CH19-54	16.1	44.37	80.20
	CH18-52A1	13.0	35.83	
WEST NORDEAU - 1	CH22-60	4.6	11.32	80.50
	CH22-64AW	6.0	14.77	
	CH22-64A	5.5	13.54	
	NW08-06	8.5	20.93	
	NW08-07	8.1	19.94	
WEST NORDEAU - 2	CH23-91	16.0	45.03	80.50
	CH22-63	7.0	19.70	

Composite	Hole ID	Length (m)	Weight (kg)	Overall Weight (kg)
	CH22-63	5.6	15.76	

The main composite includes material from all drill core intervals, whereas the two variability composites were prepared to cover a range of gold grades. Table 13-3 presents the gold grades obtained by metallic screening on a 1000 g subsample screened at 75 µm.

Table 13-3: 2026 Composite gold grade

Composite	Gold grade (g/t)
CHIMO	3.86
CHIMO - 1	2.83
CHIMO - 2	4.88
EAST CHIMO	4.07
EAST CHIMO - 1	5.26
EAST CHIMO - 2	2.88
WEST NORDEAU	3.41
WEST NORDEAU - 1	3.91
WEST NORDEAU - 2	2.91

The main composite sample and the three variability samples were submitted to mineralogical analysis. Table 13-4 shows the result.

Table 13-4: 2026 Composite mineralogical composition

Minerals (%)	CHIMO - 1 (%)	CHIMO - 2 (%)	EAST CHIMO - 1 (%)	EAST CHIMO - 2 (%)	WEST NORDEAU - 1 (%)	WEST NORDEAU - 2 (%)
Copper Sulphides	<0.1	<0.1	<0.1	0.1	<0.1	<0.1
Cuprite/Native Copper	0.0	0.0	0.0	0.0	0.0	<0.1
Pyrite	0.1	0.2	0.8	0.4	0.1	0.2
Pyrrhotite	1.6	2.0	2.4	2.7	1.0	1.1
Arsenopyrite	2.5	1.0	1.6	1.3	2.6	3.1
CoFeNiAsS	<0.1	<0.1	0.0	0.0	<0.1	0.0
Other Sulphide Minerals	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Iron Oxides	0.1	0.1	0.1	0.2	0.2	0.2
Quartz	14.6	14.2	26.1	32.1	26.3	27.1
Feldspars	53.6	32.7	49.7	35.1	28.3	22.0
Chlorite	5.2	7.1	4.8	5.9	4.6	6.4
Micas	3.9	10.9	0.9	6.6	4.6	6.3
Pyroxene/Amphibole	12.9	19.4	6.3	7.9	23.6	14.8
Epidote	0.5	3.8	0.4	1.4	0.9	1.4



Minerals (%)	CHIMO - 1 (%)	CHIMO - 2 (%)	EAST CHIMO - 1 (%)	EAST CHIMO - 2 (%)	WEST NORDEAU - 1 (%)	WEST NORDEAU - 2 (%)
Calcium Carbonates	2.1	3.5	3.7	2.3	2.4	4.9
Titanium Minerals	1.5	1.6	2.3	2.0	1.6	2.1
Apatite	0.3	0.2	0.2	0.3	0.1	0.4
Tourmaline	0.5	2.1	0.1	1.0	3.2	9.2
Others	0.5	1.1	0.8	0.8	0.6	0.8
Total	100.0	99.9	100.0	100.0	100.0	100.0

Mineralogical characterization indicates that pyrrhotite content is relatively higher in the Chimo and East Chimo composites, whereas the West Nordeau composites contain a greater proportion of arsenopyrite. The variability composites exhibit limited intra-deposit variation, supporting the representativeness of the master composites.

Sulphide contents are considered moderate, and no deleterious minerals or elevated concentrations of base metal sulphides were identified. Consequently, the mineralogical data do not indicate any significant impediments to gold recovery by conventional cyanide leaching, although the presence of pyrrhotite and arsenopyrite may increase cyanide and oxygen consumption during processing.

13.2.3 Metallurgical Test Work

13.2.3.1 Grindability

The comminution characterization program included Bond Ball Mill Work Index (BWi) and SMC tests on the three master composites representing the Chimo, East Chimo, and West Nordeau deposits. The Bond Work Index results were used to assess the resistance of each sample to ball milling, while the SMC parameters were used to evaluate sample competency and crushing characteristics. Table 13-5 and Table 13-6 summarize the test results.

Table 13-5 : 2026 Composite Grindability results

Sample ID	Bond Ball Mill Work Index kWh/t
CHIMO	13.1
EAST CHIMO	14.3
WEST NORDEAU	13.1



Table 13-6 : 2026 Main Composite SMC Result

Sample ID	A x b	Mia kWh/t	Mib kWh/t	Mic kWh/t	SCSE kWh/t
CHIMO	25	27.8	17.9	11.8	13.1
EAST CHIMO	32	23.2	19.6	9.4	11.3
WEST NORDEAU	31	24.0	17.7	9.9	11.9

Figure 13-2 : SCSE Results Compared to JKSimMEt Database

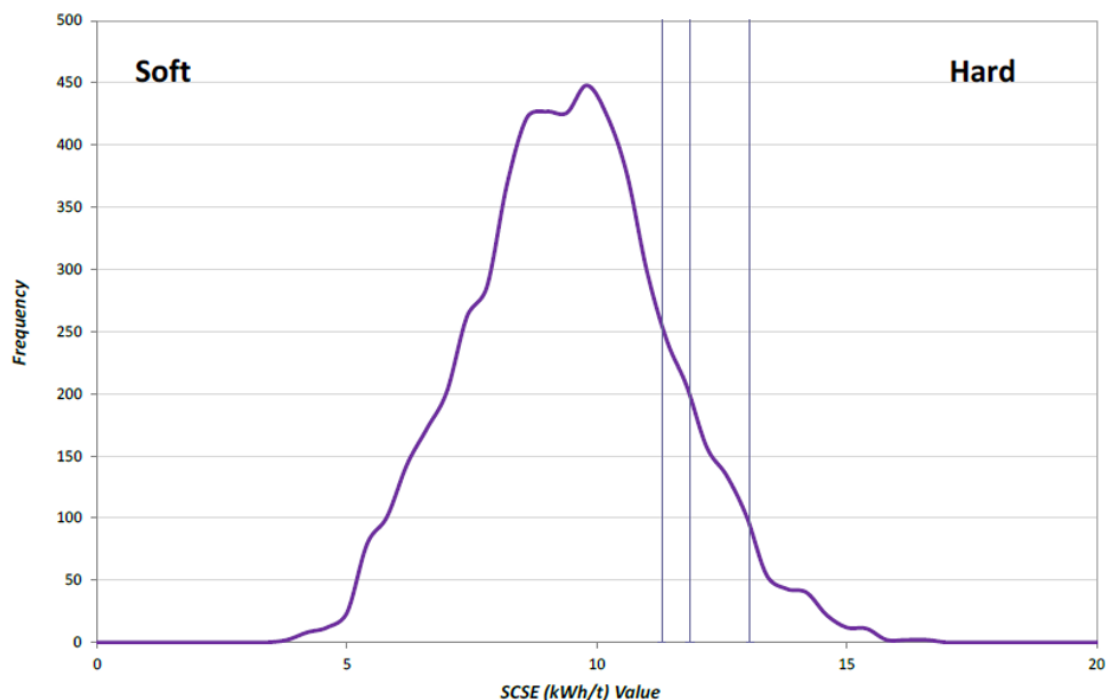


Figure 13-2 compares the SCSE (SAG Circuit Specific Energy) values for the master composites with those in the JKTech database. The Chimo, East Chimo, and West Nordeau composites generally plot in the upper portion of the distribution, indicating moderate to high competency relative to the reference database. Although the results suggest somewhat elevated energy requirements for a SAG-based comminution circuit, all three deposits exhibit similar comminution characteristics and are expected to perform comparably in a conventional crushing and grinding circuit.

13.2.3.2 Gravity Extraction

Gravity concentration was evaluated for each master composite. A 10 kg charge was ground to a nominal K80 of 55 μm , corresponding to the design grind size, and processed in a single pass through



a laboratory-scale Knelson concentrator using a 100 g cone, fluidization water at 3 psi, a centrifugal force of 60 G, and a processing time of 19.5 minutes. The Knelson concentrate was upgraded on a Mozley table. The table tailings were then combined with the Knelson tailings to produce the gravity tailings, which were subsequently leached as described in Section 13.2.3.3. Two gravity tests were conducted on each composite sample. Table 13-7 summarizes the gravity test results and presents the average values for each composite.

Table 13-7 : 2026 Main Composite Gravity Test Results

Sample	Product	Weigh yield		Gold Assay	Gold distribution
		%	g	g/t	%
CHIMO	Table concentrate	0.6	59.1	337.3	61.4
	Gravity Tails	99.4	9941.0	1.2	38.6
	Calculated Feed	100.0	10000.0	3.1	100.0
	Measured Feed			4.5	
EAST CHIMO	Table concentrate	0.6	58.4	271.7	43.9
	Gravity Tails	99.4	9941.6	1.9	56.1
	Calculated Feed	100.0	10000.0	3.5	100.0
	Measured Feed			3.9	
WEST NORDEAU	Table concentrate	0.8	78.4	187.7	56.1
	Gravity Tails	99.2	9921.6	1.1	43.9
	Calculated Feed	100.0	10000.0	2.6	100.0
	Measured Feed			3.1	

Gravity concentration recovered between 43.9% and 61.4% of the gold contained in the tested samples. The Chimo sample achieved the highest gravity recovery, at 61.4%, while the East Chimo and West Nordeau samples achieved recoveries of 43.9% and 56.1%, respectively. These results indicate the presence of a significant free-gold component and demonstrate that a significant proportion of the gold is liberated at the test grind size. Because the Knelson concentrate was upgraded on a Mozley table, the resulting concentrate is not expected to contain a significant proportion of arsenopyrite-hosted gold that would be unavailable for dissolution. Therefore, the installation of gravity recovery equipment should be considered to recover liberated gold early in the process and minimize the accumulation of coarse gold within the grinding circuit and associated slurry-handling equipment.



13.2.3.3 Cyanidation

13.2.3.3.1 Whole Rock Cyanidation

Whole-rock cyanide leach tests were conducted on six variability composites and three master composites. Under standard conditions, three tests were completed on each variability composite at three grind sizes spanning a K80 range of 48 µm to 128 µm to establish the sensitivity of gold recovery to grind size. Additional tests were then completed at the finest grind size to assess the effects of modified leach conditions: lead nitrate addition, pre-aeration, a combination of both, and elevated pH. The combined lead nitrate addition and pre-aeration condition, which was retained as the basis for the conceptual design, was also applied to the three master composites. A total of thirty-two (32) tests were completed.

Each leach test was conducted over a retention time of 48 to 72 hours, with solution samples collected after 3, 6, 24, and 48 hours and, where applicable, after 72 hours to establish gold dissolution kinetics. Tests on the variability composites were conducted over 72 hours. Once the dissolution kinetics were shown to be essentially complete after 48 hours, the tests on the master composites were terminated at that point. Unless otherwise stated, all tests were conducted under the following conditions:

- Cyanide concentration maintained at 500 ppm NaCN;
- pH maintained at 11.0;
- Bottle left open to the atmosphere as the oxygen source.

Table 13-8 shows the results obtained from the cyanidation testwork:

Table 13-8 : 2026 Composites Whole Rock Cyanide Leach Results

Cyanidation feed			Gold Grade			Recovery	Reagent Consumption	
Sample ID	Conditions	F80	Calculated feed	Measured feed	Tailings grade	Gold	NaCN	Ca(OH) ₂
		(µm)	(g/t)	(g/t)	(g/t)	(%)	(kg/t)	(kg/t)
Chimo - 1	Standard	48	3.9	3.3	0.200	94.8	0.95	0.94
Chimo - 1	Standard	95	2.8	3.3	0.285	90.0	0.47	0.78
Chimo - 1	Standard	97	3.2	3.3	0.350	89.0	0.51	1.03
Chimo - 2	Standard	52	6.0	5.6	0.120	98.0	0.78	0.57
Chimo - 2	Standard	95	3.9	5.6	0.285	92.8	0.49	1.02
Chimo - 2	Standard	128	3.7	5.6	0.450	87.7	0.33	1.08
Chimo - 2	Standard	128	4.2	5.6	0.650	84.6	0.34	0.72
East Chimo - 1	Standard	52	3.6	4.4	0.360	90.1	1.17	1.32
East Chimo - 1	Standard	88	4.8	4.4	0.510	89.5	0.58	1.13
East Chimo - 1	Standard	122	3.7	4.4	0.610	83.5	0.47	1.23

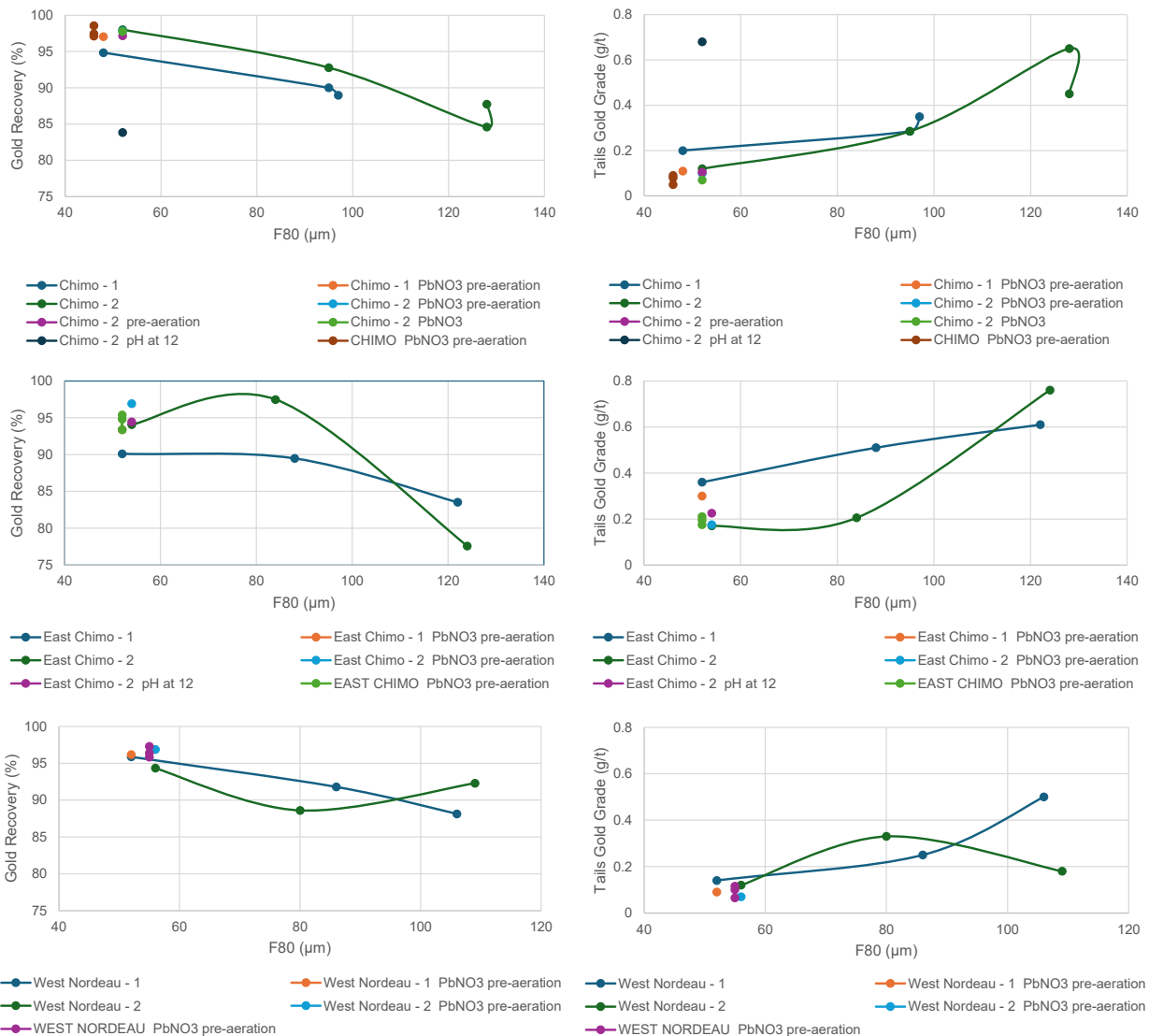


Cyanidation feed			Gold Grade			Recovery	Reagent Consumption	
Sample ID	Conditions	F80	Calculated feed	Measured feed	Tailings grade	Gold	NaCN	Ca(OH) ₂
		(µm)	(g/t)	(g/t)	(g/t)	(%)	(kg/t)	(kg/t)
East Chimo - 2	Standard	54	2.9	3.4	0.170	94.1	0.86	0.84
East Chimo - 2	Standard	84	8.1	3.4	0.205	97.5	0.76	1.32
East Chimo - 2	Standard	124	3.4	3.4	0.760	77.6	0.53	0.84
West Nordeau - 1	Standard	52	3.4	3.7	0.140	95.9	0.88	0.81
West Nordeau - 1	Standard	86	3.0	3.7	0.250	91.8	0.28	0.89
West Nordeau - 1	Standard	106	4.2	3.7	0.500	88.1	0.28	0.96
West Nordeau - 2	Standard	56	2.1	2.6	0.120	94.3	0.89	1.12
West Nordeau - 2	Standard	80	2.9	2.6	0.330	88.6	0.38	1.10
West Nordeau - 2	Standard	109	2.3	2.6	0.180	92.3	0.34	1.04
Chimo - 1	PbNO ₃ + pré-aération	48	3.7	3.3	0.110	97.0	0.21	0.80
Chimo - 2	PbNO ₃ + pré-aération	52	4.6	5.6	0.100	97.8	0.22	0.87
Chimo - 2	PbNO ₃	52	3.1	5.6	0.070	97.8	0.70	1.08
Chimo - 2	Pré-aération 16 h	52	3.7	5.6	0.105	97.2	0.27	1.05
East Chimo - 1	PbNO ₃ + pré-aération	52	4.5	4.4	0.300	93.4	0.26	0.83
East Chimo - 2	PbNO ₃ + pré-aération	54	5.7	3.4	0.175	96.9	0.24	1.07
West Nordeau - 1	PbNO ₃ + pré-aération	52	2.4	3.7	0.090	96.2	0.19	0.65
West Nordeau - 2	PbNO ₃ + pré-aération	56	2.2	2.6	0.070	96.9	0.26	0.91
CHIMO	PbNO ₃ + pré-aération	46	3.5	4.5	0.090	97.4	0.32	0.44
EAST CHIMO	PbNO ₃ + pré-aération	52	3.8	3.9	0.195	94.8	0.26	0.54
WEST NORDEAU	PbNO ₃ + pré-aération	55	2.8	3.1	0.100	96.4	0.23	0.25
Chimo - 2	pH 12	52	4.2	5.6	0.680	83.8	0.55	3.27
East Chimo - 2	pH 12	54	4.0	3.4	0.225	94.4	0.79	3.64

Based on the results, several plots were prepared. Figure 13-3 shows recovery (calculated recovery) and tailings grade as a function of feed size (F80).



Figure 13-3 : Composites Cyanide Leach Recovery

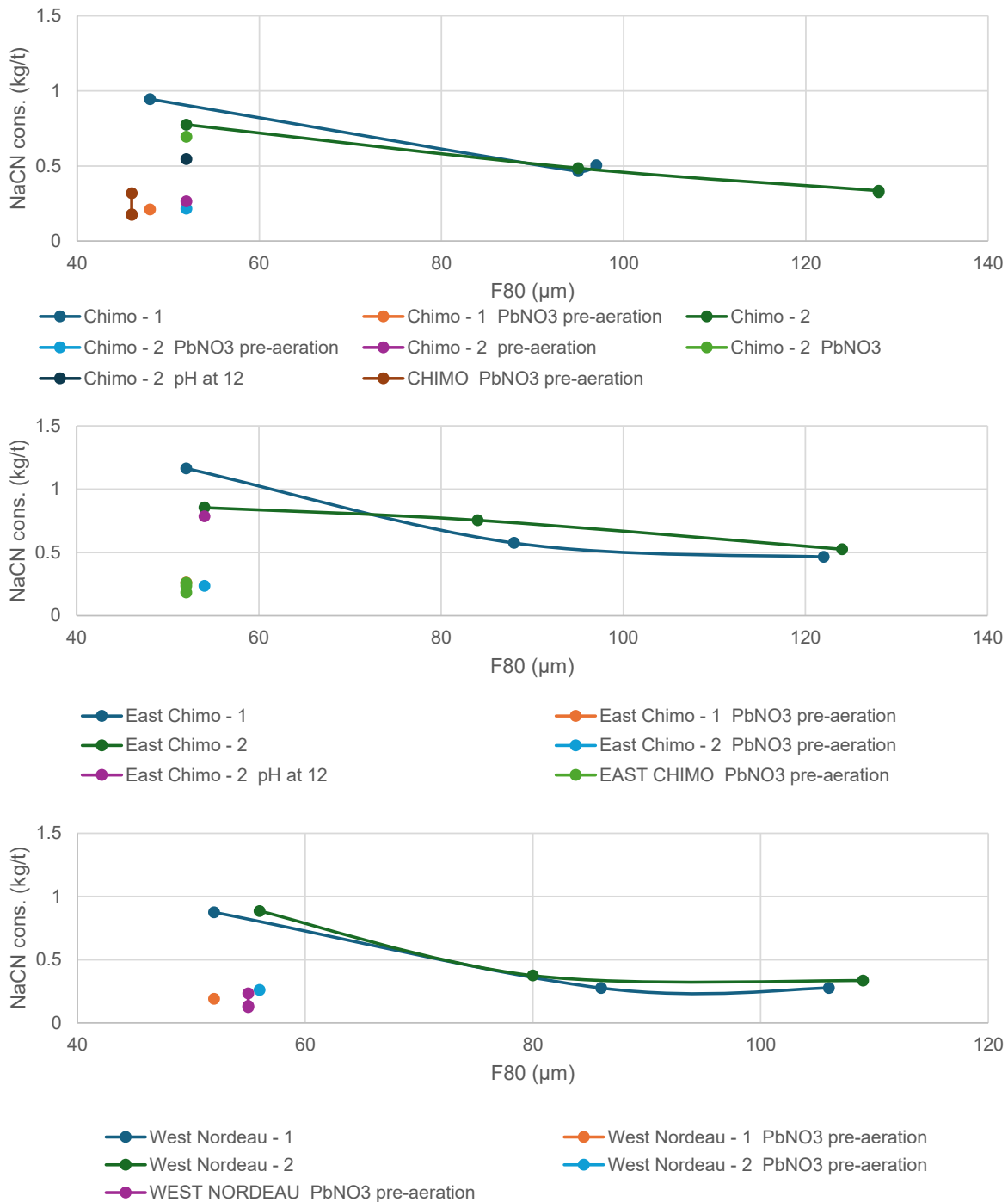


Both values are presented because variations in feed grade, caused by coarse, liberated gold, affect the calculated recovery. In contrast, the gold reporting to the tailings consists mainly of fine gold, which is more evenly distributed among the samples. The tailings grade therefore provides a clearer indication of the effect of K80, without bias from feed-grade variability.

Figure 13-4 shows the sodium cyanide consumption in function of the K80 for the different composite.



Figure 13-4 : Composites NaCN consumption



Cyanide consumption increases as grind size decreases, consistent with the greater reactive surface area of cyanide-consuming sulphide minerals. Sensitivity to grind size varies among the composites: East Chimo-1 and Chimo-1 show a marked increase in cyanide consumption at finer grind sizes, whereas Chimo-2, East Chimo-2, and both West Nordeau composites are less affected.

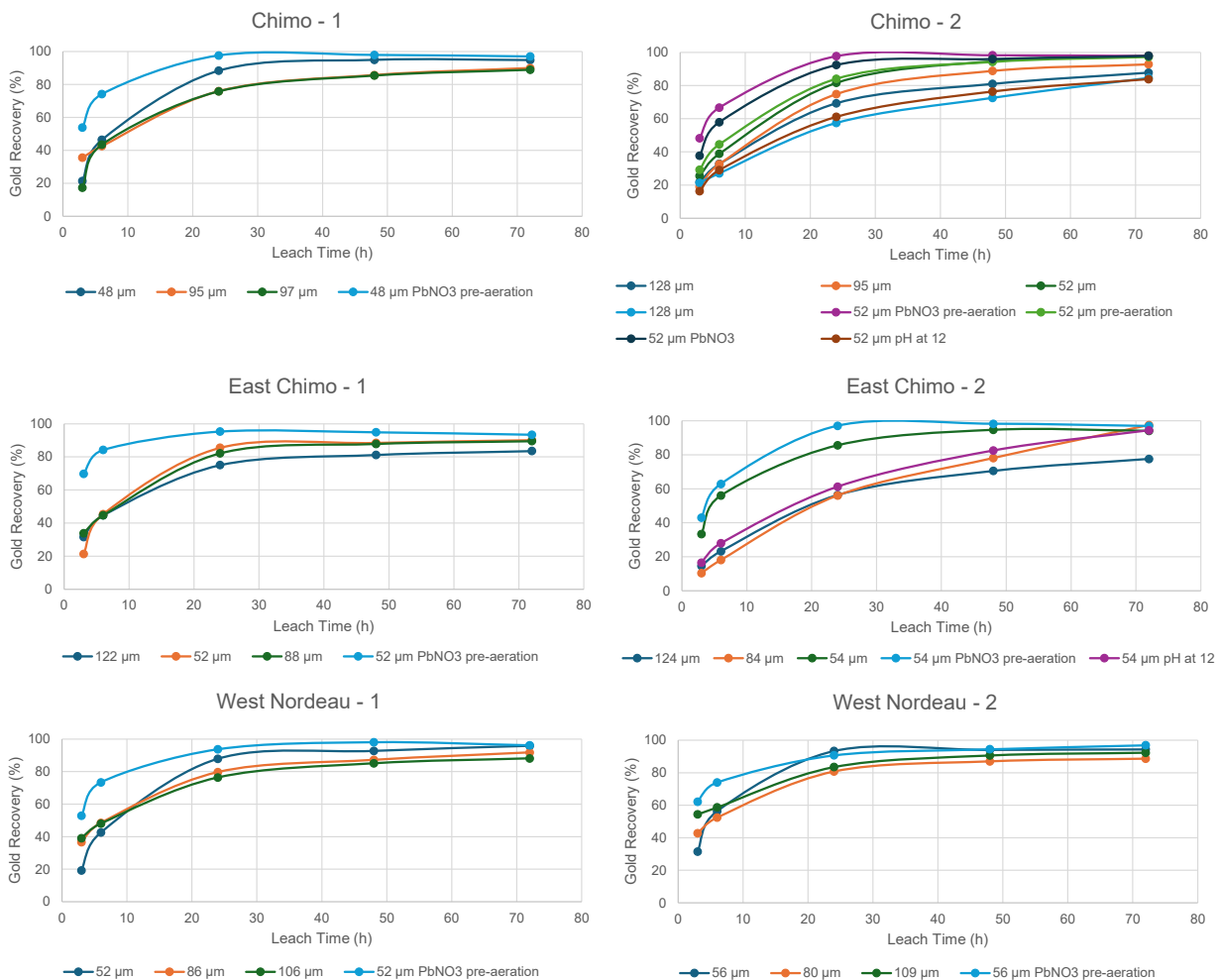


At the finest grind size tested, cyanide consumption ranges from 0.78 kg/t to 1.17 kg/t.

Lead nitrate addition combined with pre-aeration reduces cyanide consumption at the same grind size for all composites from the three deposits. Lead nitrate addition alone does not affect cyanide consumption, whereas pre-aeration alone has a similar effect to the combined treatment. Lead nitrate precipitates soluble sulphide species, while pre-aeration oxidizes reactive sulphides before cyanide addition; together, these mechanisms also normalize the response among the composites. The cyanide consumption used in the operating cost estimate is based on a grind size of 50µm.

Figure 13-5 present the kinetic gold dissolution for all the tests:

Figure 13-5 : Composites Cyanide Leach Kinetics



Beyond 24 hours of leaching, gold extraction is consistently higher at finer grind sizes for all six composites. After 24 hours, extraction ranges from 82% to 93% at a K80 of 48 µm to 56 µm,



compared with 56% to 75% at a K80 above 120 µm; at the coarser grind sizes, leaching remains incomplete after 72 hours. During the initial hours of leaching, however, no consistent relationship with grind size is observed, and several composites show higher early extraction at the coarser grind sizes.

Lead nitrate addition combined with pre-aeration markedly accelerates gold dissolution across all six composites. Between 43% and 70% of the gold is extracted within the first three hours, and 91% to 98% is extracted within 24 hours, representing essentially all of the final extraction. Beyond 24 hours, the extraction curves are flat, whereas under standard conditions at the same grind size, leaching continues for up to 48 hours or longer. The Chimo-2 sample was selected to isolate the individual effects of pre-aeration and lead nitrate addition. Pre-aeration alone does not significantly affect leaching kinetics, whereas lead nitrate addition improves them. However, combining pre-aeration with lead nitrate results in faster kinetics than using lead nitrate alone. This combined condition was tested on all composites, with similar improvements observed for each.

13.2.3.4 Gravity Tails Cyanidation

Following the gravity test, the gravity tailings were subjected to cyanide leaching under the same conditions used for the other cyanidation tests. Table 13-9 presents the results of this test together with those of the two leach tests conducted on the main composite. The tailings grade is substantially lower than the grades obtained in some previous tests conducted without a gravity circuit.

Table 13-9 : 2026 Gravity Tails Cyanide Leach Results

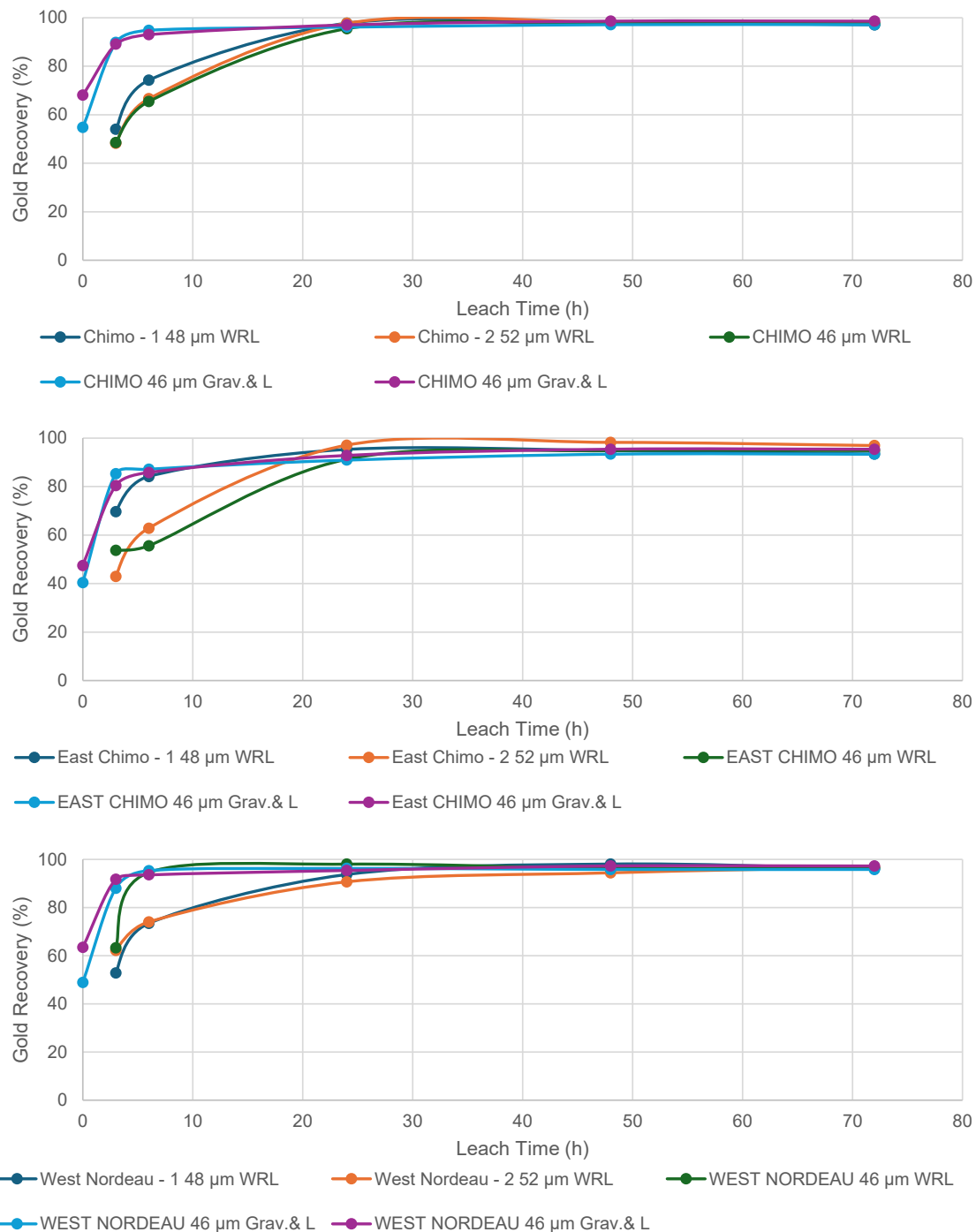
Cyanidation feed		Gold Grade			Recovery	Reagent consumption	
Sample	F80 (µm)	Calculated feed (g/t)	Measured feed (g/t)	Tailings (g/t)	Gold (%)	NaCN (kg/t)	Ca(OH) ₂ (kg/t)
CHIMO	46	2.8	4.5	0.080	97.1	0.17	0.75
CHIMO	46	3.4	4.5	0.050	98.5	0.18	0.51
EAST CHIMO	52	3.2	3.9	0.210	93.4	0.23	1.01
EAST CHIMO	52	3.8	3.9	0.175	95.4	0.18	0.64
WEST NORDEAU	55	2.7	3.1	0.115	95.8	0.13	0.65
WEST NORDEAU	55	2.4	3.1	0.065	97.3	0.12	0.43

Figure 13-6 compares the kinetics of the gravity tails leach tests with the whole-rock leach tests. In this figure, overall recovery includes the gold recovered by gravity; therefore, recovery starts at 77%



at time zero.

Figure 13-6 : Main Composite Whole Rock and Gravity tails Kinetics



Gold dissolution kinetics are faster for the gravity tailings because coarse gold particles, which dissolve more slowly, are removed in the gravity concentrate. The results also show that overall gold



recovery remains essentially unchanged. This comparison further suggests that the leaching kinetics of material produced by a grinding circuit operating in closed circuit with hydrocyclones would be faster than those indicated by the laboratory tests, in which coarse gold fragments are not further reduced in size.

13.2.4 Environmental Testing

The final cyanidation test on the gravity tailings generated approximately 5 kg of cyanide tailings slurry. This slurry was used in a cyanide destruction test to confirm the process's ability to reduce cyanide concentrations in the tailings. The resulting solution and solids were collected for chemical analysis. The solids were also submitted for environmental leach testing and acid-base accounting.

13.2.4.1 Cyanide Destruction

Laboratory cyanide destruction testing was conducted as a continuous process to mimic plant operation, including copper addition. This approach was used to confirm reagent dosages. Table 13-10 presents the liquid chemistry results for the feed and discharge.

Table 13-10 : Main Composite cyanide destruction results

Sample	SO ₂ Dosage gSO ₂ /gCN _{wad}	CuSO ₄ ·5H ₂ O Dosage mg/L	Stream	CN _{wad} mg/L	Cu mg/L	Fe mg/L	Ni mg/L	Zn mg/L
CHIMO	4.7	24.9	Feed	280	14.30	1.80	1.46	0.60
			Discharge	4.35	17.50	3.50	1.50	0.30
CHIMO	8.5	10.0	Feed	260	11.60	0.50	1.52	1.60
			Discharge	1.80	0.02	12.20	0.90	0.05
EAST CHIMO	6.1	26.2	Feed	270	9.00	0.40	0.42	1.80
			Discharge	55.00	13.60	1.80	0.30	1.10
EAST CHIMO	9.9	96.0	Feed	530	7.80	0.80	0.27	4.00
			Discharge	0.08	0.02	4.60	0.02	0.05
WEST NORDEAU	5.9	57.9	Feed	290	1.57	0.10	0.13	0.05
			Discharge	81.50	17.30	7.50	0.50	0.20
WEST NORDEAU	7.9	10.0	Feed	270	3.11	0.30	0.13	0.30
			Discharge	0.22	0.02	1.40	0.13	0.05

The first three cyanide destruction tests were performed with a SO₂ dosage of about 5 gSO₂/gCN_{wad} and the reduction of CN_{wad} was not enough. Three more tests were then conducted with higher SO₂ and CuSO₄ dosages and allows to reduce the CN_{wad} concentration below 2ppm. It shows that the process could reduce the CN_{wad} concentration below acceptable levels when the



SO₂ dosage is high enough.

13.2.5 Recovery Prediction

Based on the available testwork, gold recovery can be estimated from key feed characteristics. Regression analysis indicates that recovery correlates with F80, feed gold grade and feed sulphur grade. The resulting equation is:

$$Rec = 110.6 - 0.15 \times F80 + 0.85 \times Gold - 8.83 \times Sulphur$$

Where:

- > Rec: Gold recovery in %
- > F80: The F80, dimension of the 80% undersize content in µm
- > Gold: The gold grade of the feed in g/t
- > Sulphur: The sulfur assay of the feed in %

The three variables used in the equation are based on linear regression, and all three show a statistically significant effect on gold recovery. However, the equation should be used with caution and is only valid within the ranges tested. The applicable ranges are:

- > F80: From 46 to 128 µm
- > Gold: From 2.11 to 8.10 g/t
- > Sulphur: From 0.93 to 1.46 %

The value predicted by this equation represents the gold recovered to solution and is based on whole-rock leach tests. A circuit incorporating gravity recovery is assumed to achieve a similar overall gold recovery because the gravity concentrate is subsequently cyanide leached, thereby accounting for the gold that would otherwise report to the gravity concentrate, including any non-soluble gold. To estimate final plant gold recovery, a 0.5% reduction should be applied to account for gold losses in the solution exiting the recovery circuit.

13.2.6 Testwork Results Interpretation

The testwork indicates that direct cyanide leaching would have slower kinetics and higher cyanide consumption than leaching with a pre-aeration step and lead nitrate addition. The costs associated with those steps are substantially lower than the resulting metallurgical benefit. The ideal flowsheet would therefore include the pre-aeration step. Because no cyanide would be added to the grinding circuit, a gravity gold recovery circuit would help reduce gold accumulation in the grinding circuit.



The tests also confirm that gold recovery is influenced by the product size distribution down to a P80 of 50 μm . A finer grind cannot currently be recommended because no test has confirmed its effect on gold recovery, and the available test results do not allow the grinding cost for a finer grind to be estimated accurately. The improved recovery at a finer grind, combined with the presence of arsenopyrite in the resource samples, indicates that some of the fine gold could be disseminated in the arsenopyrite. In that case, concentrating sulphides from the mill feed by flotation and regrinding the flotation concentrate could provide better or equivalent recovery with a significant reduction in grinding costs. The current results do not allow such a gain to be predicted, but targeted new testwork would provide the necessary information.

The cyanide destruction test shows that the SO_2 dosage required to complete tailings detoxification should be higher than dosages observed elsewhere. The results suggest that an $\text{SO}_2/\text{CN}_{\text{wad}}$ ratio of 8 could be adequate.

13.2.7 Future Testwork

The extent of the testwork is appropriate for a PEA-level study. Additional testwork will be required for the next stage of pre-feasibility or feasibility. The following work should be planned:

- Settling tests for thickener design
- Filtration tests on tailings if underground backfill is implemented
- Adding more variability samples to improve design robustness
- Preparation of a tailings sample for environmental assessment

Because interpretation of the results indicates that adding a flotation step with concentrate regrinding could improve metallurgical performance, a test program would be required to verify this assumption. Such a test should first be performed on a composite sample to determine the process criteria. The following approach is suggested:

- Determine the optimal grinding size distribution before flotation tests
- Perform a flotation test to produce a sulphide concentrate
- Determine the grind-recovery curve of the resulting concentrate
- Determine the grinding work index to reach the optimal size reduction
- Apply the treatment to variability samples

Such a test would allow the gold recovery and treatment cost associated with the flotation flowsheet to be determined. The process flowsheet, including process equipment sizing, could then be



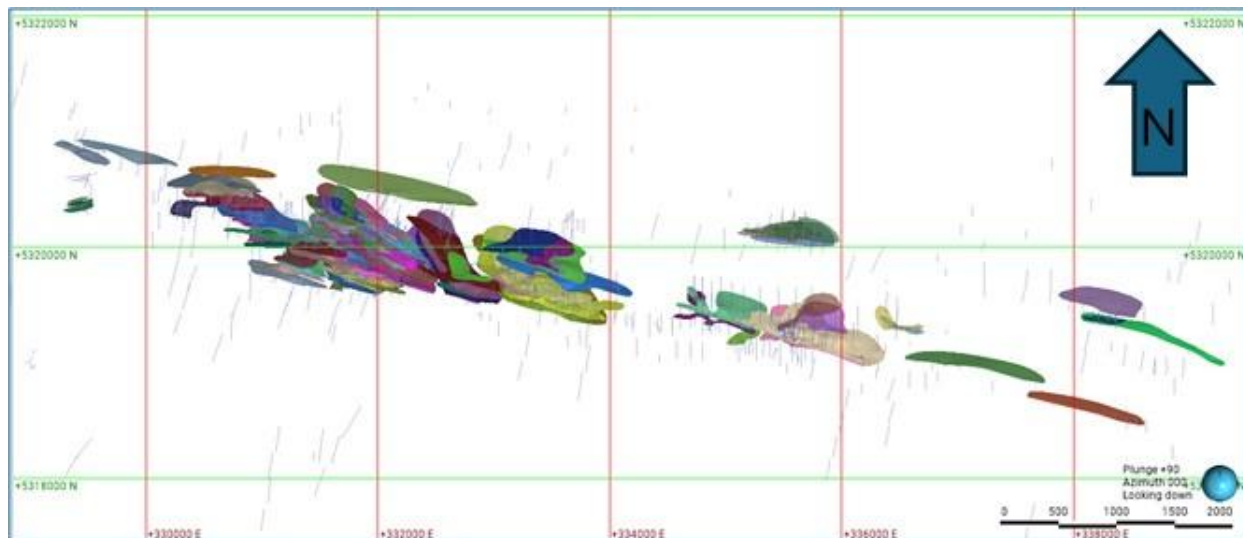
designed.



14 MINERAL RESOURCE ESTIMATE

The mineral resource estimate herein (the “2025 MRE” or the “MRE”) covers the Cadillac Project. Figure 14-1 shows the Cadillac Project in Plan view.

Figure 14-1 : Overall Plan view of the modelled zones on the Cadillac Project



14.1 Methodology

Leapfrog Geo™ and Edge™ v.2025.1.1 (“Leapfrog”) was used to update the mineralized zones and to generate the drill hole intercepts for each solid. Leapfrog was used for the compositing, 3D block modelling, grade interpolation, and classification. Statistical studies were conducted using Excel, Leapfrog, and Supervisor. The pit optimization to develop the mineral resource-constraining pit shells was done using the pseudoflow algorithm in Deswik software. The stope optimization to develop the underground mineral resource was done using Deswik.SO software. The methodology for the mineral resource estimation involved the following steps:

- Database verification and validation;
- 3D modelling update of the mineralized zones;
- Drill hole intercepts and composite generation;
- Basic statistics;
- Capping;
- Geostatistical analysis including variography;



- Block modelling and grade interpolation;
- Block model validation;
- Mineral resource classification;
- Cut-off grade calculation;
- Pit shell optimization;
- DSO optimization;
- Preparation of the mineral resource statement.

14.2 Resource Database

The MRE zones are based on 4,477 drill holes, totalling 535,360 meters drilled and 167,978 assays.

The cut-off date for the drill hole database was February 17, 2025.

The resource database was validated, and protocols were reviewed before proceeding to the resource estimation. The validation steps are detailed in Chapter 12 of this Report.

The QP is of the opinion that the database is appropriate for the purposes of the mineral resource estimation and that the sample density, quality and spatial distribution allow to make a reliable estimate of the geometry, tonnage and grade continuity of the mineralization in accordance with the level of confidence established by the mineral resource categories as set forth in the CIM Standards.

14.3 Mineralized Zones Model

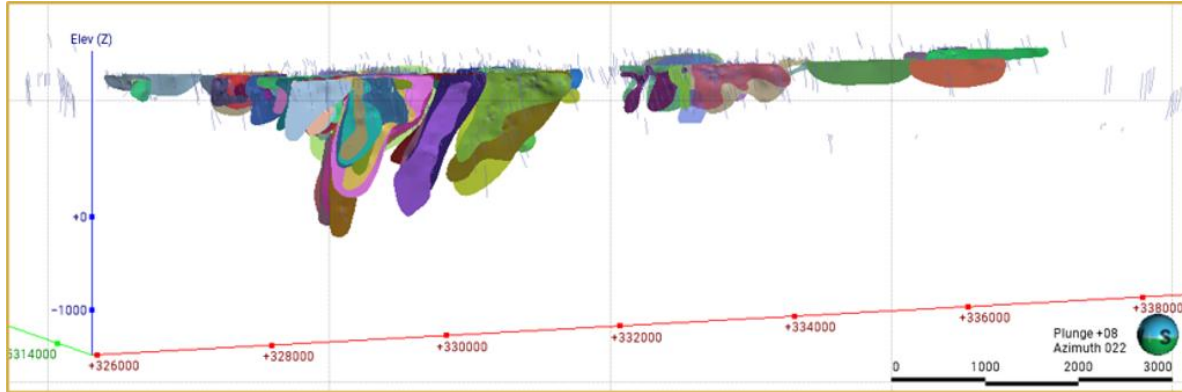
Based on Cartier's preliminary modeling work, a total of 86 high-grade domains and 98 lower-grade envelopes were modelled, as was the overburden and the topography for the purpose of this MRE. They were modelled using geological knowledge of the deposit, grade continuity, and geological information provided in the DDH (i.e., lithology, alteration, and structure). The high-grade mineralized zones and the lower-grade envelopes were clipped to the overburden/bedrock interface.

The QP reviewed the geological model in 3D view, plan view and cross-section and is of the opinion that the level of detail to which the geology model was constructed represents adequately the complexity of the deposit. In the QP's opinion, the geological model is appropriate for the size, grade distribution and geometry of the mineralized zones and is suitable for the resource estimation of the Project.

Figure 14-2 shows a 3D view of the geological model.



Figure 14-2 : View of the Mineralized zones of the Cadillac Project (slight plunge towards North-Northeast)



14.3.1 Voids Model

Blocks affected by depleted historical underground workings were sterilized.

14.3.2 Overburden and Topography

A surface derived from collars was used for the topographic surface. The overburden-rock interface was created using drill holes collar coordinates, elevation, and lithological description.

14.4 Compositing

All raw assay data intersecting the mineralized zones were assigned individual rock codes. These coded intercepts were used to produce basic statistics on sample lengths and grades. A total of 42,690 assays are included in the high-grade mineralized zones, and 56,810 in the lower-grade envelopes.

Compositing drill hole samples aimed to homogenize the database for statistical analysis and remove any bias associated with sample lengths that may exist in the original database. The composite length was determined using original sample length statistics and the general thickness of the mineralized zones. Compositing was done within each individual domain making sure that composite samples do not cross domain boundaries. Unsourced intervals within mineralized zones were forced to 0 g/t Au.

The average sample length is 1.00 m and the median is 0.99 m inside the high-grade domains. In the envelopes, the average sample length is 0.99 m and the median is 1.00 m. Based on these statistics and geological considerations, 54,137 composites were generated in the high-grade zones and 21,575 in the lower-grade envelopes, with an average length of 1 m after redistributing the tails.



14.5 Capping

It is common practice to statistically examine the higher grades within a population and to trim them to a lower grade value based on the results of a statistical study. Capping is performed on high-grade values considered to be outliers. An outlier is an observation that appears inconsistent with most of the data. High-grade capping was done on the composited assay.

The capping values were defined by checking for abnormal breaks or changes in the slope on the grade distribution probability plot while making sure that the coefficient of variation of the capped data was ideally lower than 2.00 and no more than 10% of the total contained metal was enclosed within the first 1% of the highest-grade samples. The use of various statistical methods allows for a selection of the capping threshold in a more objective and justified manner.

Basic statistics for Au composites and capped composites are summarized in Table 14-1 and Table 14-2. Figure 14-3 and Figure 14-4 show graphs supporting the capping threshold decisions for one of the high-grade zones and one of the lower-grade envelopes. This exercise was completed for each individual zone.

Based on individual statistical study for each zone, composites were capped at a grade varying from 5 g/t to 110 g/t Au for the high-grade zones, and at a grade varying from 1 g/t to 5 g/t Au for the lower-grade envelopes.





Table 14-1: Basic statistics on composites and high-grade capping values for Au for the high-grade zones

Zones	Count	Uncapped				Capping	Capped Count	Capped Percent	Metal Loss	Capped			
		Max	Mean	Median	COV					Max	Mean	Median	COV
Ch_1A	516	22.5	3.2	2.08	1.18	20	2	0.39%	0.22%	20	3.2	2.08	1.17
Ch_1B	558	73	2.2	0.92	1.97	20	2	0.36%	1.52%	20	2.1	0.92	1.52
Ch_2	1,369	170.89	5.12	1.2	2.44	95	7	0.51%	1.34%	95	5.06	1.2	2.34
Ch_3	973	273.5	6.31	2.1	2.61	90	8	0.82%	4.94%	90	5.87	2.1	2.07
Ch_4B	101	88.58	2.19	0.24	4.7	5	5	4.95%	66.83%	5	0.72	0.24	1.68
Ch_4C	83	9.6	0.87	0.3	1.65	5	1	1.20%	6.58%	5	0.81	0.3	1.42
Ch_5A	29	8.2	1.26	0.6	1.35	5	1	3.45%	4.91%	5	1.15	0.6	1.14
Ch_5B	21,663	438.8	3.02	1.4	2.3	110	7	0.03%	1.07%	110	2.99	1.4	1.97
Ch_5C	2,493	37.37	1.26	0.01	2.42	30	4	0.16%	0.54%	30	1.25	0.01	2.38
Ch_5D	87	28.28	1.34	0.5	2.61	10	2	2.30%	25.10%	10	1.08	0.5	1.78
Ch_5M	4,064	48.89	2.15	0.97	1.69	45	3	0.07%	0.10%	45	2.14	0.97	1.68
Ch_5M2	9,009	223.7	2.28	0.8	2.55	90	5	0.06%	1.32%	90	2.25	0.8	2.26
Ch_5N	4,501	175.7	1.58	0.33	3.05	40	5	0.11%	4.24%	40	1.51	0.33	2.29
Ch_6	1,101	91.75	2.99	1.56	1.82	30	5	0.45%	4.18%	30	2.87	1.56	1.42
Ch_6B	562	108.29	2.72	1.13	2.26	30	2	0.36%	5.75%	30	2.58	1.13	1.69
Ch_6C	251	32.95	1.29	0.4	2.28	10	5	1.99%	10.58%	10	1.17	0.4	1.76
Ch_6N1	372	52.7	2.2	1.03	2.3	20	4	1.08%	10.99%	20	1.95	1.03	1.62
Ch_6P	244	81.13	3.5	1.25	2.39	30	4	1.64%	13.71%	30	3.05	1.25	1.78
Ch_6P2	80	15.42	2.79	1.4	1.22	10	5	6.25%	5.64%	10	2.63	1.4	1.11
Ch_7	18	8.49	1.45	0.9	1.36	5	1	5.56%	8.58%	5	1.26	0.9	1.07
Ch_S	24	6.66	1.58	1.26	0.9	5	1	4.17%	4.57%	5	1.52	1.26	0.8
EB_A	13	4.13	1.29	1.22	1.07	5	0	0.00%	0.00%	4.13	1.29	1.22	1.07
EB_B	39	20.19	2.51	0.98	1.69	15	1	2.56%	5.69%	15	2.37	0.98	1.57
EB_C	49	60.14	2.97	0.3	3.12	15	3	6.12%	27.20%	15	1.86	0.3	2.1
EC_2B	26	20.68	1.28	0.28	3.13	5	1	3.85%	50.98%	5	0.67	0.28	1.87
EC_3	85	44.71	2.6	0.9	2.68	15	2	2.35%	23.88%	15	1.9	0.9	1.65





Zones	Count	Uncapped				Capping	Capped Count	Capped Percent	Metal Loss	Capped			
		Max	Mean	Median	COV					Max	Mean	Median	COV
EC_4B	130	26.7	2.41	1.17	1.84	15	3	2.31%	10.49%	15	2.15	1.17	1.49
EC_5B	256	64.83	2.28	1.23	2.13	10	7	2.73%	14.96%	10	1.94	1.23	1.21
EC_5C	54	19.95	1.63	0.63	1.94	15	1	1.85%	6.65%	15	1.54	0.63	1.73
EC_5M2	225	55.71	2.38	1.29	1.85	10	6	2.67%	12.10%	10	2.1	1.29	1.11
EN_1	52	2	0.25	0.01	1.86	5	0	0.00%	0.00%	2	0.25	0.01	1.86
EN_2	24	46.4	4.24	1.28	2.18	10	1	4.17%	43.10%	10	2.72	1.28	1.17
EN_3	65	9.7	0.87	0.3	1.86	10	0	0.00%	0.00%	9.7	0.87	0.3	1.86
EN_4	205	129.99	3.36	1.32	3.04	30	2	0.98%	18.59%	30	2.76	1.32	1.65
EN_4B	6	3.41	1.27	0.95	0.85	5	0	0.00%	0.00%	3.41	1.27	0.95	0.85
EN_5	173	20.4	1.83	0.7	1.68	20	1	0.58%	0.15%	20	1.83	0.7	1.67
NC_1	46	7.39	1.34	0.88	1.16	10	0	0.00%	0.00%	7.39	1.34	0.88	1.16
NC_2	44	3.86	1.45	1.32	0.73	5	0	0.00%	0.00%	3.86	1.45	1.32	0.73
NC_3	106	28.27	3.06	1.99	1.27	20	1	0.94%	2.68%	20	2.98	1.99	1.16
No_1	65	53.4	3.26	0.69	2.45	20	2	3.08%	17.45%	20	2.52	0.69	1.69
No_3	68	38.41	1.81	0.24	3.23	10	2	2.94%	41.59%	10	1.1	0.24	1.9
No_4	21	10.79	1.48	0.7	1.63	10	1	4.76%	2.76%	10	1.44	0.7	1.57
Po_1	12	4.16	2.01	1.64	0.42	5	0	0.00%	0.00%	4.16	2.01	1.64	0.42
Po_2	17	6.5	1.08	0.74	1.49	10	0	0.00%	0.00%	6.5	1.08	0.74	1.49
Po_3	44	21.53	1.98	0.7	1.9	15	1	2.27%	8.62%	15	1.83	0.7	1.66
Po_4	63	18.53	2.82	1.43	1.34	15	2	3.17%	1.11%	15	2.76	1.43	1.29
Po_VG31	66	134.85	3.19	0.39	5.16	15	1	1.52%	59.66%	15	1.37	0.39	1.98
SB_1	21	10.1	2.01	0.71	1.39	10	1	4.76%	0.29%	10	2	0.71	1.39
SB_2	17	9.16	1.59	0.69	1.36	10	0	0.00%	0.00%	9.16	1.59	0.69	1.36
WC_1A	19	25.73	3.85	0.24	1.86	10	2	10.53%	37.50%	10	2.4	0.24	1.37
WC_1B	21	39.96	3.17	0.99	2.63	5	1	4.76%	55.40%	5	1.5	0.99	0.96
WC_2	276	183.85	5.06	1.86	2.8	90	2	0.72%	7.48%	90	4.72	1.86	2.23
WC_3	76	95.31	3.03	0.19	5.02	5	3	3.95%	80.49%	5	0.61	0.19	1.98
WC_3B	161	25.7	1.49	0.34	2.1	10	3	1.86%	10.37%	10	1.31	0.34	1.55





Zones	Count	Uncapped				Capping	Capped Count	Capped Percent	Metal Loss	Capped			
		Max	Mean	Median	COV					Max	Mean	Median	COV
WC_4B	1,051	38.36	2.38	1.53	1.46	30	2	0.19%	0.48%	30	2.37	1.53	1.42
WC_5A	22	25.73	3.92	0.6	1.63	20	1	4.55%	6.49%	20	3.66	0.6	1.52
WC_5B	74	7.9	0.54	0.01	2.04	5	1	1.35%	3.99%	5	0.5	0.01	1.72
WC_5N	382	88.48	2.11	0.77	2.77	30	2	0.52%	8.42%	30	1.94	0.77	2.04
WC_6	43	13	2.53	1.87	0.82	15	0	0.00%	0.00%	13	2.53	1.87	0.82
WC_6C	6	30	6.3	1.81	1.69	30	0	0.00%	0.00%	30	6.3	1.81	1.69
WC_6N1	11	3.91	1.38	0.81	0.94	5	0	0.00%	0.00%	3.91	1.38	0.81	0.94
WC_6P	16	10.37	3.19	1.62	1.01	10	1	6.25%	0.46%	10	3.17	1.62	1.01
WC_6P2	24	6.5	1.7	1.44	0.73	10	0	0.00%	0.00%	6.5	1.7	1.44	0.73
WC_7	20	7.5	2.62	1.68	0.78	10	0	0.00%	0.00%	7.5	2.62	1.68	0.78
We_1	21	7.87	0.99	0.73	1.63	10	0	0.00%	0.00%	7.87	0.99	0.73	1.63
We_2	52	1.94	0.43	0.14	1.27	5	0	0.00%	0.00%	1.94	0.43	0.14	1.27
WN_3	113	10.03	0.83	0.04	2.05	5	5	4.42%	10.88%	5	0.73	0.04	1.77
WN_3B	43	8.83	1.03	0.32	1.79	10	0	0.00%	0.00%	8.83	1.03	0.32	1.79
WN_4B	226	45.5	2.06	0.92	2.17	15	4	1.77%	9.48%	15	1.79	0.92	1.53
WN_5B	157	19.65	1.4	0.65	1.65	10	2	1.27%	4.35%	10	1.33	0.65	1.42
WN_5N	484	89	2.66	1.56	1.87	25	1	0.21%	5.08%	25	2.53	1.56	1.27
WN_A	67	20.73	1.24	0.25	2.97	5	3	4.48%	48.10%	5	0.66	0.25	1.68
WN_R	26	5.26	0.81	0.27	1.71	5	1	3.85%	1.28%	5	0.8	0.27	1.7
WN_VG10	120	173.64	6.39	0.81	2.96	45	3	2.50%	20.11%	45	4.87	0.81	1.93
WN_VG9	48	25.27	1.96	0.64	2.19	15	2	4.17%	12.67%	15	1.73	0.64	1.85
WP_1	7	3.77	1.17	0.45	1.04	5	0	0.00%	0.00%	3.77	1.17	0.45	1.04
WP_2	9	9.06	2.76	1.91	0.95	10	0	0.00%	0.00%	9.06	2.76	1.91	0.95
WP_3	14	49.69	4.62	1.2	2.71	5	1	7.14%	73.76%	5	1.43	1.2	0.89
WS_5A	9	13.55	3.7	1.87	1.16	15	0	0.00%	0.00%	13.55	3.7	1.87	1.16
WS_5B	146	38.98	2.27	1.02	1.99	20	1	0.68%	6.62%	20	2.14	1.02	1.71
WS_5C	55	16.78	1.54	0.62	2.01	5	2	3.64%	29.25%	5	1.13	0.62	1.21
WS_E	69	32	2.68	0.78	1.98	20	1	1.45%	7.31%	20	2.51	0.78	1.79





Zones	Count	Uncapped				Capping	Capped Count	Capped Percent	Metal Loss	Capped			
		Max	Mean	Median	COV					Max	Mean	Median	COV
WS_F	84	24.8	2.1	0.48	2.04	20	2	2.38%	2.18%	20	2.02	0.48	1.93
WS_H	57	8.05	1.12	0.69	1.42	10	0	0.00%	0.00%	8.05	1.12	0.69	1.42
WS_I	27	24.6	2.58	1.24	1.79	10	1	3.70%	15.03%	10	2.04	1.24	1.11
WS_J	11	8.18	1.54	0.8	1.44	10	0	0.00%	0.00%	8.18	1.54	0.8	1.44





Table 14-2: Basic statistics on composites and high-grade capping values for Au for the lower-grade envelopes

Zones	Count	Uncapped				Capping	Capped Count	Capped Percent	Metal Loss	Capped			
		Max	Mean	Median	COV					Max	Mean	Median	COV
Ch	10,417	63.27	0.43	0.01	4.32	5.00	155.00	1.49%	22.27%	5.00	0.33	0.01	2.36
EB	185	35.65	0.60	0.10	5.32	2.50	4.00	2.16%	55.21%	2.50	0.27	0.10	1.89
EC	2,244	51.49	0.51	0.16	4.08	5.00	31.00	1.38%	21.54%	5.00	0.40	0.16	1.95
EN	1,100	8.91	0.29	0.12	2.35	5.00	6.00	0.55%	3.65%	5.00	0.28	0.12	2.09
NC	382	25.93	0.47	0.22	3.54	2.50	8.00	2.09%	25.91%	2.50	0.35	0.22	1.30
No	278	4.07	0.33	0.08	1.84	2.50	6.00	2.16%	5.27%	2.50	0.31	0.08	1.65
Po	686	5.14	0.22	0.03	2.02	2.50	5.00	0.73%	2.94%	2.50	0.22	0.03	1.86
SB	109	1.29	0.20	0.20	1.01	1.00	2.00	1.83%	2.25%	1.00	0.20	0.20	0.92
WC	2,298	119.91	0.56	0.09	5.06	5.00	37.00	1.61%	17.44%	5.00	0.46	0.09	2.00
We	235	22.26	0.23	0.03	6.39	1.00	2.00	0.85%	41.25%	1.00	0.14	0.03	1.49
WN	2,789	62.38	0.35	0.20	3.81	5.00	9.00	0.32%	8.15%	5.00	0.32	0.20	1.60
WP	57	7.89	0.76	0.28	1.72	2.50	4.00	7.02%	0.22%	2.50	0.62	0.28	1.29
WS	795	11.42	0.30	0.20	2.13	2.50	7.00	0.88%	3.59%	2.50	0.28	0.20	1.44



Figure 14-3 : Graphs and Stats supporting Au capping on composites in the high-grade zone Ch_5B

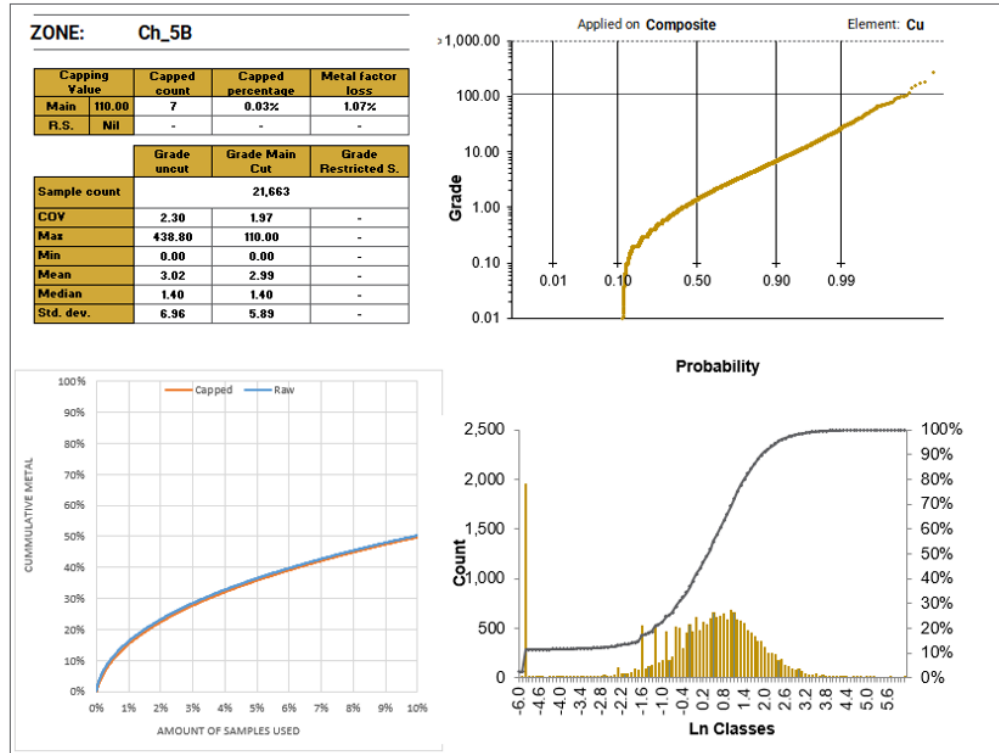
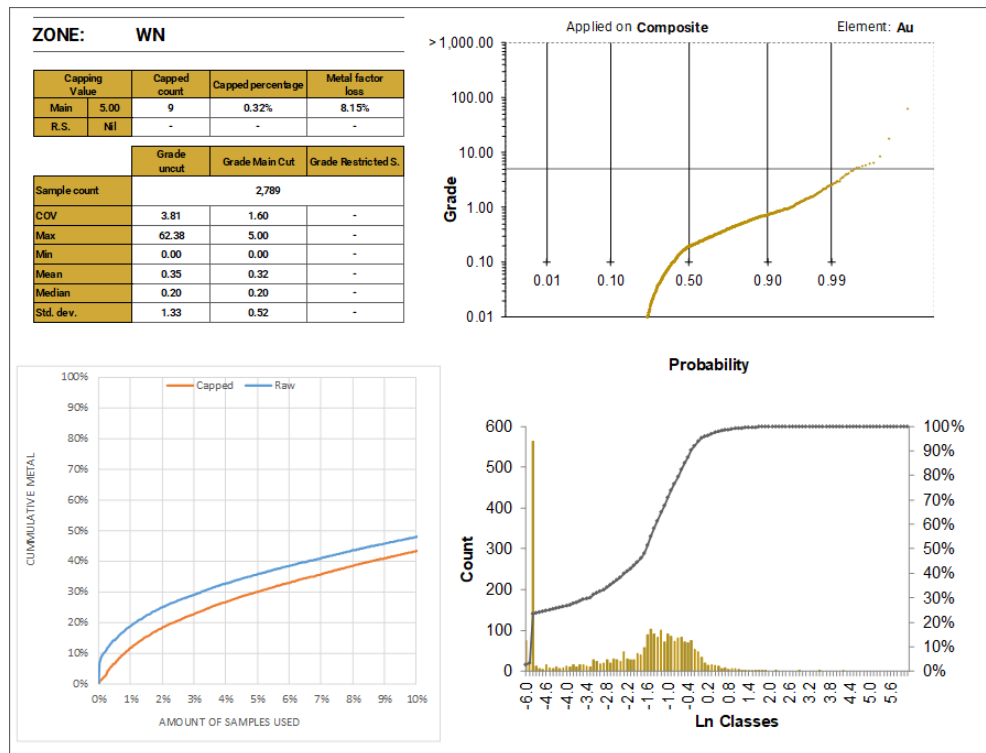


Figure 14-4 : Graphs and Stats supporting Au capping on composites in the lower-grade zones WN



14.6 Density

Bulk density is an important parameter used to estimate tonnages for the estimated volumes derived from the grade block model.

A total of 316 density measurements were collected by Cartier within the mineralized zones, 306 of which were added in 2025 for this MRE. The samples selected were from a variety of lithologies located across the Project and also included a range of associated gold grades. The specific gravity (SG) measurement was determined by pycnometer method. A summary of the SG data is presented in Table 14-3.

Table 14-3 : Density basic statistics

Zones	Units	Count	Min	Max	Average	Median
Ch	g/cm3	171	2.70	3.08	2.87	2.86
EB	g/cm3	16	2.70	2.89	2.80	2.81
EC	g/cm3	47	2.70	3.09	2.84	2.83
NC	g/cm3	43	2.66	3.00	2.88	2.89
WC	g/cm3	8	2.78	2.94	2.81	2.80
WN	g/cm3	31	2.68	3.03	2.86	2.89

For this MRE, fixed density values were established on a per zone basis, corresponding to the average of the SG data. A fixed density of 2.86 was assigned to country rock. A fixed density of 2.00 g/cm3 was assigned to the overburden.

14.7 Variogram Analysis and Search Ellipsoids

A semi-variogram is a common tool used to measure the spatial variability within a zone. Typically, samples taken far apart will vary more than samples taken close to each other. A variogram gives a measure of how much two samples taken from the same mineralized zone will vary in grade depending on the distance between those samples, and therefore allowing building search ellipsoids to be used during interpolation.

Three-dimensional directional variography was carried out on the composites using the Snowden Supervisor software. Variograms were modelled in the three orthogonal directions to define a 3D ellipsoid for each domain. The three directions of ellipsoid axes were set by using the variogram fans



and visually confirmed with geological knowledge of the deposit. Lag distances were set according to drill hole grid spacing specific to the structural domain analyzed.

Then, a mathematical model was interpreted to best-fit the shape of the calculated variogram for each direction. Three components were defined for the mathematical model: the nugget effect, the sill, and the range.

All variography tests were modelled with a nugget effect, as determined from the downhole semi-variograms and two spherical structures.

It was not possible to yield robust variography parameters for five of the groups of zones (No, Po, We, Sb, and Wp). Variography from a nearby group sharing similar orientation and dip was applied in these cases.

Table 14-4 presents the chosen variogram model parameters. Figure 14-5 illustrates an example of the variography results for one of the zones.

In the QP's opinion, the data density and spatial distribution of this project are adequate to produce acceptable experimental variograms to which models can be fitted with confidence.



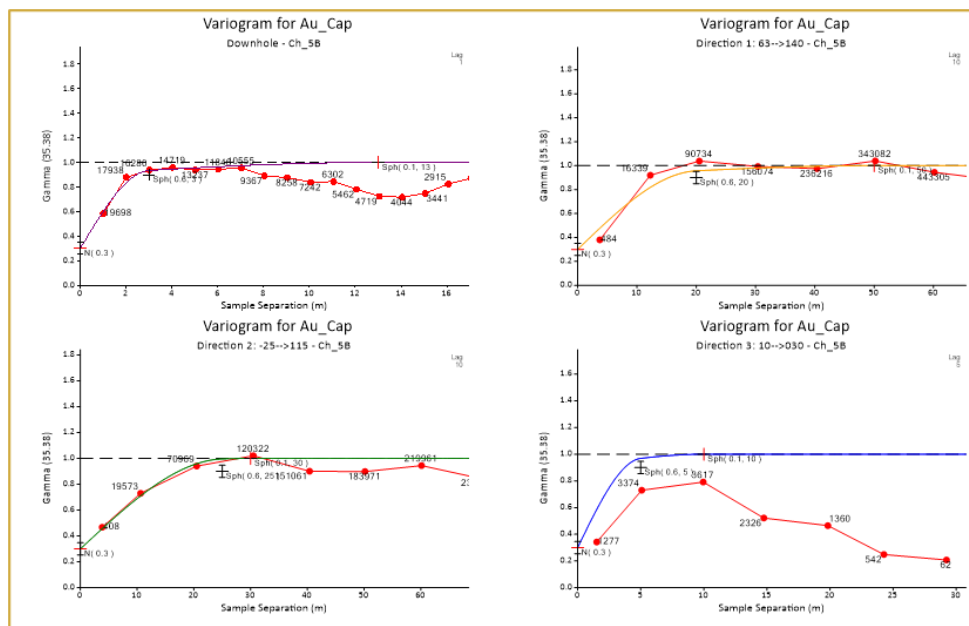


Table 14-4: Variogram model parameters

			First structure				Second structure				Leapfrog orientation (ZXZ)		
Zones	Element	Nugget	Sill	Range X (m)	Range Y (m)	Range Z (m)	Sill	Range X (m)	Range Y (m)	Range Z (m)	Dip Azimuth	Dip	Pitch
Ch_5B	Au	0.300	0.600	20	25	5	0.100	50	30	10	Variable Orientation		
Ch_5C	Au	0.050	0.800	15	25	2	0.150	40	30	5	Variable Orientation		
Ch_5M	Au	0.200	0.550	25	45	2	0.250	100	90	5	Variable Orientation		
Ch_5M2	Au	0.150	0.600	25	20	5	0.250	50	45	10	Variable Orientation		
Ch_5N	Au	0.250	0.550	15	15	5	0.200	40	20	15	Variable Orientation		
All other Ch zones	Au	0.300	0.600	20	25	5	0.100	50	30	10	Variable Orientation		
WC	Au	0.300	0.250	25	25	5	0.450	100	60	80	Variable Orientation		
EC	Au	0.350	0.100	45	30	5	0.550	100	85	10	Variable Orientation		
WN	Au	0.150	0.300	45	25	5	0.550	75	60	10	Variable Orientation		
EN	Au	0.300	0.050	50	40	5	0.650	80	60	10	Variable Orientation		
NC	Au	0.300	0.050	60	40	5	0.650	100	60	10	Variable Orientation		
WS	Au	0.150	0.300	30	25	5	0.550	50	30	10	Variable Orientation		
No	Au	0.150	0.300	45	25	5	0.550	75	60	10	Variable Orientation		
Po	Au	0.150	0.300	30	25	5	0.550	50	30	10	Variable Orientation		
We	Au	0.150	0.300	30	25	5	0.550	50	30	10	Variable Orientation		
Eb	Au	0.300	0.220	45	40	5	0.480	90	60	10	Variable Orientation		
Sb	Au	0.300	0.050	50	40	5	0.650	80	60	10	Variable Orientation		
Wp	Au	0.150	0.300	30	25	5	0.550	50	30	10	Variable Orientation		



Figure 14-5 : Variography study for high-grade zone Ch_5B



14.8 Block Model

The block model was constructed in Leapfrog for the current mineral resource estimate using the block model parameters provided in Table 14-5. Individual block cells have dimensions of 5 m long (X-axis) by 5 m wide (Y-axis) by 5 m vertical (Z-axis). The size of the blocks was chosen to best match the drilling pattern, the thickness of the zones, the complexity of the geological model, and plausible future mining methods. The block size was discussed with engineers working on the Project.

The block model was coded using the octree sub-block method, down to 0.625 m, reflecting the proportion of each solid inside every block. All blocks falling within a solid were assigned the corresponding solid block code. Table 14-6 shows the various attributes in the block model.

Table 14-5: Block model parameters

Properties	X (column)	Y (row)	Z (level)
Origin coordinates	328,800	5,318,200	450
Number of blocks	2140	600	360
Block size (m)	5	5	5
Sub-block size (down to)	0.625	0.625	0.625
Rotation	0		



Table 14-6: Block model coding

Attribute	Description
Blockcode	Blockcode per zone; country rock, overburden, air
Density	Density value (fixed)
Au_CAP_OK	Au capped grade interpolated with Ordinary Kriging
Classification_Final	3 = Measured 4 = Indicated 5 = Inferred 6 = Exploration Target

14.9 Search Ellipsoid Strategy

The ranges and orientation of the ellipsoids used for the interpolation were established using the variography study. Other interpolation parameters are derived from combining kriging neighbourhood analyses and the QP's professional experience.

Based on geostatistical analysis and general geological knowledge of the Project, the ranges of the ellipsoids correspond to the range of the variogram for the first pass and twice the range of the variogram for the second pass.

It should be mentioned that the classification was mostly based on geological confidence, grade continuity and drill hole spacing. For this reason, some interpolated blocks could not be classified as either Inferred or Indicated or measured. Refer to the Mineral Resource Classification section further below for more details.

Table 14-7 and Table 14-8 present the orientation and range of the search ellipsoids for each pass.





Table 14-7 : Search ellipsoids range and orientation by interpolation passes for HG Zones

Zone	Element	Leapfrog orientation (ZXZ)			First Pass			Second Pass		
		Dip	Azimuth	Pitch	Range X (m)	Range Y (m)	Range Z (m)	Range X (m)	Range Y (m)	Range Z (m)
Ch_1A	Au	Variable Orientation			50	30	10	100	60	20
Ch_1B	Au	Variable Orientation			50	30	10	100	60	20
Ch_2	Au	Variable Orientation			50	30	10	100	60	20
Ch_3	Au	Variable Orientation			50	30	10	100	60	20
Ch_4B	Au	Variable Orientation			50	30	10	100	60	20
Ch_4C	Au	Variable Orientation			50	30	10	100	60	20
Ch_5A	Au	Variable Orientation			50	30	10	100	60	20
Ch_5B	Au	Variable Orientation			50	30	10	100	60	20
Ch_5C	Au	Variable Orientation			40	30	10	80	60	20
Ch_5D	Au	Variable Orientation			50	30	10	100	60	20
Ch_5M	Au	Variable Orientation			100	90	10	200	180	20
Ch_5M2	Au	Variable Orientation			50	45	10	100	90	20
Ch_5N	Au	Variable Orientation			40	20	15	80	60	30
Ch_6	Au	Variable Orientation			50	30	10	100	60	20
Ch_6B	Au	Variable Orientation			50	30	10	100	60	20
Ch_6C	Au	Variable Orientation			50	30	10	100	60	20
Ch_6N1	Au	Variable Orientation			50	30	10	100	60	20
Ch_6P	Au	Variable Orientation			50	30	10	100	60	20
Ch_6P2	Au	Variable Orientation			50	30	10	100	60	20
Ch_7	Au	Variable Orientation			50	30	10	100	60	20
Ch_S	Au	Variable Orientation			50	30	10	100	60	20
EB_A	Au	Variable Orientation			90	60	10	180	120	20
EB_B	Au	Variable Orientation			90	60	10	180	120	20
EB_C	Au	Variable Orientation			90	60	10	180	120	20
EC_2B	Au	Variable Orientation			100	85	10	200	170	20
EC_3	Au	Variable Orientation			100	85	10	200	170	20
EC_4B	Au	Variable Orientation			100	85	10	200	170	20





Zone	Element	Leapfrog orientation (ZXZ)			First Pass			Second Pass		
		Dip	Azimuth	Pitch	Range X (m)	Range Y (m)	Range Z (m)	Range X (m)	Range Y (m)	Range Z (m)
EC_5B	Au	Variable Orientation			100	85	10	200	170	20
EC_5C	Au	Variable Orientation			100	85	10	200	170	20
EC_5M2	Au	Variable Orientation			100	85	10	200	170	20
EN_1	Au	Variable Orientation			80	60	10	160	120	20
EN_2	Au	Variable Orientation			80	60	10	160	120	20
EN_3	Au	Variable Orientation			80	60	10	160	120	20
EN_4	Au	Variable Orientation			80	60	10	160	120	20
EN_4B	Au	Variable Orientation			80	60	10	160	120	20
EN_5	Au	Variable Orientation			80	60	10	160	120	20
NC_1	Au	Variable Orientation			100	60	10	200	120	20
NC_2	Au	Variable Orientation			100	60	10	200	120	20
NC_3	Au	Variable Orientation			100	60	10	200	120	20
No_1	Au	Variable Orientation			75	60	10	150	120	20
No_3	Au	Variable Orientation			75	60	10	150	120	20
No_4	Au	Variable Orientation			75	60	10	150	120	20
Po_1	Au	Variable Orientation			50	30	10	100	60	20
Po_2	Au	Variable Orientation			50	30	10	100	60	20
Po_3	Au	Variable Orientation			50	30	10	100	60	20
Po_4	Au	Variable Orientation			50	30	10	100	60	20
Po_VG31	Au	Variable Orientation			50	30	10	100	60	20
SB_1	Au	Variable Orientation			90	60	10	180	120	20
SB_2	Au	Variable Orientation			90	60	10	180	120	20
WC_1A	Au	Variable Orientation			100	60	10	200	120	20
WC_1B	Au	Variable Orientation			100	60	10	200	120	20
WC_2	Au	Variable Orientation			100	60	10	200	120	20
WC_3	Au	Variable Orientation			100	60	10	200	120	20
WC_3B	Au	Variable Orientation			100	60	10	200	120	20
WC_4B	Au	Variable Orientation			100	60	10	200	120	20
WC_5A	Au	Variable Orientation			100	60	10	200	120	20





Zone	Element	Leapfrog orientation (ZXZ)			First Pass			Second Pass		
		Dip	Azimuth	Pitch	Range X (m)	Range Y (m)	Range Z (m)	Range X (m)	Range Y (m)	Range Z (m)
WC_5B	Au	Variable Orientation			100	60	10	200	120	20
WC_5N	Au	Variable Orientation			100	60	10	200	120	20
WC_6	Au	Variable Orientation			100	60	10	200	120	20
WC_6C	Au	Variable Orientation			100	60	10	200	120	20
WC_6N1	Au	Variable Orientation			100	60	10	200	120	20
WC_6P	Au	Variable Orientation			100	60	10	200	120	20
WC_6P2	Au	Variable Orientation			100	60	10	200	120	20
WC_7	Au	Variable Orientation			100	60	10	200	120	20
We_1	Au	Variable Orientation			50	30	10	100	60	20
We_2	Au	Variable Orientation			50	30	10	100	60	20
WN_3	Au	Variable Orientation			75	60	10	150	120	20
WN_3B	Au	Variable Orientation			75	60	10	150	120	20
WN_4B	Au	Variable Orientation			75	60	10	150	120	20
WN_5B	Au	Variable Orientation			75	60	10	150	120	20
WN_5N	Au	Variable Orientation			75	60	10	150	120	20
WN_A	Au	Variable Orientation			75	60	10	150	120	20
WN_R	Au	Variable Orientation			75	60	10	150	120	20
WN_VG10	Au	Variable Orientation			75	60	10	150	120	20
WN_VG9	Au	Variable Orientation			75	60	10	150	120	20
WP_1	Au	Variable Orientation			50	30	10	100	60	20
WP_2	Au	Variable Orientation			50	30	10	100	60	20
WP_3	Au	Variable Orientation			50	30	10	100	60	20
WS_5A	Au	Variable Orientation			50	30	10	100	60	20
WS_5B	Au	Variable Orientation			50	30	10	100	60	20
WS_5C	Au	Variable Orientation			50	30	10	100	60	20
WS_E	Au	Variable Orientation			50	30	10	100	60	20
WS_F	Au	Variable Orientation			50	30	10	100	60	20
WS_H	Au	Variable Orientation			50	30	10	100	60	20
WS_I	Au	Variable Orientation			50	30	10	100	60	20





Zone	Element	Leapfrog orientation (ZXZ)			First Pass			Second Pass		
		Dip Azimuth	Dip	Pitch	Range X (m)	Range Y (m)	Range Z (m)	Range X (m)	Range Y (m)	Range Z (m)
WS_J	Au	Variable Orientation			50	30	10	100	60	20

Table 14-8 : Search ellipsoids range and orientation by interpolation passes for LG Zones

Zone	Element	Leapfrog orientation (ZXZ)			First Pass			Second Pass		
		Dip Azimuth	Dip	Pitch	Range X (m)	Range Y (m)	Range Z (m)	Range X (m)	Range Y (m)	Range Z (m)
Ch_1A	Au	Variable Orientation			50	30	10	100	60	20
Ch_1B	Au	Variable Orientation			50	30	10	100	60	20
Ch_2	Au	Variable Orientation			50	30	10	100	60	20
Ch_3	Au	Variable Orientation			50	30	10	100	60	20
Ch_3B	Au	Variable Orientation			50	30	10	100	60	20
Ch_4B	Au	Variable Orientation			50	30	10	100	60	20
Ch_4C	Au	Variable Orientation			50	30	10	100	60	20
Ch_5A	Au	Variable Orientation			50	30	10	100	60	20
Ch_5C	Au	Variable Orientation			40	30	10	80	60	20
Ch_5D	Au	Variable Orientation			50	30	10	100	60	20
Ch_5M	Au	Variable Orientation			100	90	10	200	180	20
Ch_5N	Au	Variable Orientation			40	20	15	80	60	30
Ch_6	Au	Variable Orientation			50	30	10	100	60	20
Ch_6B	Au	Variable Orientation			50	30	10	100	60	20
Ch_6C	Au	Variable Orientation			50	30	10	100	60	20
Ch_6N1	Au	Variable Orientation			50	30	10	100	60	20
Ch_6P	Au	Variable Orientation			50	30	10	100	60	20
Ch_6P2	Au	Variable Orientation			50	30	10	100	60	20
Ch_7	Au	Variable Orientation			50	30	10	100	60	20
Ch_R	Au	Variable Orientation			50	30	10	100	60	20
Ch_S	Au	Variable Orientation			50	30	10	100	60	20
Ch_T	Au	Variable Orientation			50	30	10	100	60	20





Zone	Element	Leapfrog orientation (ZXZ)			First Pass			Second Pass		
		Dip Azimuth	Dip	Pitch	Range X (m)	Range Y (m)	Range Z (m)	Range X (m)	Range Y (m)	Range Z (m)
EB_A	Au	Variable Orientation			90	60	10	180	120	20
EB_B	Au	Variable Orientation			90	60	10	180	120	20
EB_C	Au	Variable Orientation			90	60	10	180	120	20
EC_2	Au	Variable Orientation			100	85	10	200	170	20
EC_2B	Au	Variable Orientation			100	85	10	200	170	20
EC_3	Au	Variable Orientation			100	85	10	200	170	20
EC_3B	Au	Variable Orientation			100	85	10	200	170	20
EC_4B	Au	Variable Orientation			100	85	10	200	170	20
EC_5B	Au	Variable Orientation			100	85	10	200	170	20
EC_5C	Au	Variable Orientation			100	85	10	200	170	20
EC_5M2	Au	Variable Orientation			100	85	10	200	170	20
EC_6N1	Au	Variable Orientation			100	85	10	200	170	20
EN_1	Au	Variable Orientation			80	60	10	160	120	20
EN_2	Au	Variable Orientation			80	60	10	160	120	20
EN_3	Au	Variable Orientation			80	60	10	160	120	20
EN_4	Au	Variable Orientation			80	60	10	160	120	20
EN_4B	Au	Variable Orientation			80	60	10	160	120	20
EN_5	Au	Variable Orientation			80	60	10	160	120	20
EN_6	Au	Variable Orientation			80	60	10	160	120	20
NC_1	Au	Variable Orientation			100	60	10	200	120	20
NC_2	Au	Variable Orientation			100	60	10	200	120	20
NC_3	Au	Variable Orientation			100	60	10	200	120	20
No_1	Au	Variable Orientation			75	60	10	150	120	20
No_2	Au	Variable Orientation			75	60	10	150	120	20
No_3	Au	Variable Orientation			75	60	10	150	120	20
No_4	Au	Variable Orientation			75	60	10	150	120	20
No_5	Au	Variable Orientation			75	60	10	150	120	20
Po_1	Au	Variable Orientation			50	30	10	100	60	20
Po_2	Au	Variable Orientation			50	30	10	100	60	20





Zone	Element	Leapfrog orientation (ZXZ)			First Pass			Second Pass		
		Dip Azimuth	Dip	Pitch	Range X (m)	Range Y (m)	Range Z (m)	Range X (m)	Range Y (m)	Range Z (m)
Po_3	Au	Variable Orientation			50	30	10	100	60	20
Po_4	Au	Variable Orientation			50	30	10	100	60	20
Po_5B	Au	Variable Orientation			50	30	10	100	60	20
Po_VG31	Au	Variable Orientation			50	30	10	100	60	20
SB_1	Au	Variable Orientation			90	60	10	180	120	20
SB_2	Au	Variable Orientation			90	60	10	180	120	20
WC_1A	Au	Variable Orientation			100	60	10	200	120	20
WC_1B	Au	Variable Orientation			100	60	10	200	120	20
WC_2	Au	Variable Orientation			100	60	10	200	120	20
WC_3	Au	Variable Orientation			100	60	10	200	120	20
WC_3B	Au	Variable Orientation			100	60	10	200	120	20
WC_4B	Au	Variable Orientation			100	60	10	200	120	20
WC_5A	Au	Variable Orientation			100	60	10	200	120	20
WC_5B	Au	Variable Orientation			100	60	10	200	120	20
WC_5N	Au	Variable Orientation			100	60	10	200	120	20
WC_6	Au	Variable Orientation			100	60	10	200	120	20
WC_6C	Au	Variable Orientation			100	60	10	200	120	20
WC_6N1	Au	Variable Orientation			100	60	10	200	120	20
WC_6P	Au	Variable Orientation			100	60	10	200	120	20
WC_6P2	Au	Variable Orientation			100	60	10	200	120	20
WC_7	Au	Variable Orientation			100	60	10	200	120	20
We_1	Au	Variable Orientation			50	30	10	100	60	20
We_2	Au	Variable Orientation			50	30	10	100	60	20
WN_1	Au	Variable Orientation			75	60	10	150	120	20
WN_2	Au	Variable Orientation			75	60	10	150	120	20
WN_3	Au	Variable Orientation			75	60	10	150	120	20
WN_3B	Au	Variable Orientation			75	60	10	150	120	20
WN_4B	Au	Variable Orientation			75	60	10	150	120	20
WN_5B	Au	Variable Orientation			75	60	10	150	120	20





Zone	Element	Leapfrog orientation (ZXZ)			First Pass			Second Pass		
		Dip Azimuth	Dip	Pitch	Range X (m)	Range Y (m)	Range Z (m)	Range X (m)	Range Y (m)	Range Z (m)
WN_5N	Au	Variable Orientation			75	60	10	150	120	20
WN_A	Au	Variable Orientation			75	60	10	150	120	20
WN_R	Au	Variable Orientation			75	60	10	150	120	20
WN_VG10	Au	Variable Orientation			75	60	10	150	120	20
WN_VG9	Au	Variable Orientation			75	60	10	150	120	20
WP_1	Au	Variable Orientation			50	30	10	100	60	20
WP_2	Au	Variable Orientation			50	30	10	100	60	20
WP_3	Au	Variable Orientation			50	30	10	100	60	20
WS_5A	Au	Variable Orientation			50	30	10	100	60	20
WS_5B	Au	Variable Orientation			50	30	10	100	60	20
WS_5C	Au	Variable Orientation			50	30	10	100	60	20
WS_E	Au	Variable Orientation			50	30	10	100	60	20
WS_F	Au	Variable Orientation			50	30	10	100	60	20
WS_H	Au	Variable Orientation			50	30	10	100	60	20
WS_I	Au	Variable Orientation			50	30	10	100	60	20
WS_J	Au	Variable Orientation			50	30	10	100	60	20



14.10 Interpolation Method

The interpolation was run on a set of points extracted from the capped composited data. The block model grades were estimated using the ordinary kriging (“OK”) method (Table 14-9). Hard boundaries were applied between the mineralized zones and surrounding country rocks to prevent grades from adjacent lithologies from being used during interpolation. As a block was estimated, it was tagged with the corresponding pass number, slope of regression, kriging efficiency, number of composites used, number of drill holes used, and drill spacing.

For comparison purposes, additional grade models were generated using ID2 and NN (Table 14-9).

Table 14-9 : Interpolation methods

Interpolation Method	Discretisation	Comments
Ordinary Kriging (OK)	3 x 3 x 3	Negative weights set to zero
Inverse Distance (ID2)	3 x 3 x 3	Anisotropic using variography ellipsoids
Nearest Neighbors (NN)	3 x 3 x 3	Real NN; no anisotropy.

14.11 Interpolation Parameters

The parameters provided in Table 14-10 were chosen for the interpolation of the block model. Although the interpolation parameters are largely inspired by the KNA study, they may differ slightly to accommodate certain interpolation needs, such as having a minimum number of drill holes or avoiding smearing effects. Multiple tests were made using different interpolation parameters.

Table 14-10 : Interpolation parameters

Zones	First Pass				Second Pass			
	Min Composite	Max Composite	Max Composite per DDH	Variography ratio	Min Composite	Max Composite	Max Composite per DDH	Variography ratio
HG Zones	4	13	3	1.00	2	13	Not Limited	2.00
LG Zones	4	13	3	1.00	2	13	Not Limited	2.00

14.12 Block Model Validation

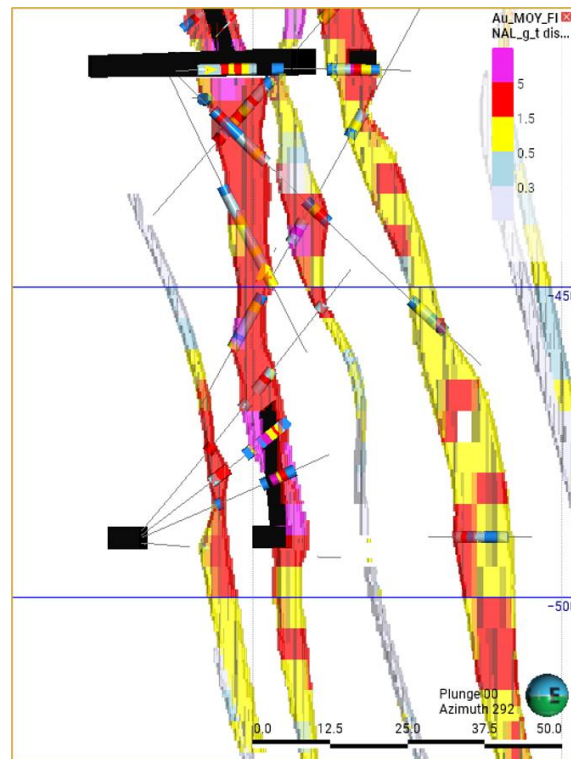
The block model was validated using several methods, including statistical analyses and a visual review of the grades in the associated drill hole. Based on these visual and statistical reviews, it is the QP’s opinion that the block model provides a reasonable estimate of in situ mineral resources.



14.12.1 Visual Validation

Block model grades were visually compared against drill hole composite grades and raw assays in cross-section (Figure 14-6), plan, longitudinal and 3D views. This visual validation process also included confirming that the proper coding was done within the various domains and checks for global and local bias. The visual comparison shows a good correlation between the values without excessive smoothing. Visual comparisons were also conducted between OK and ID2 interpolation scenarios. The OK scenario used for the mineral resource estimate produced a grade distribution honouring drill hole data and the style of mineralization observed.

Figure 14-6 : Cross Section looking NW with drillholes, assay, historical voids (black) and interpolated blocks



14.13 Mineral Resource Classification

The mineral resources were classified according to the CIM Definition Standards for Mineral Resources & Mineral Reserves published by the Canadian Institute of Mining, Metallurgy and Petroleum (“CIM Definition Standards”).

14.13.1 Mineral Resource Definition

The CIM Definition Standards clarify the following:



Inferred Mineral Resource:

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

Indicated Mineral Resource:

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

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

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

Measured Mineral Resource:

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

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

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

14.13.2 Mineral Resource Classification

The mineral resources were classified according to CIM Definition Standards. The estimated block grades were classified as either Inferred or Indicated or measured using the drill spacing, geological



continuity of mineralization, grade continuity, presence of historical mining, and overall confidence level.

Inferred Mineral Resources were defined for blocks within the mineralized zones within 50 m of a drill hole (100 m of drill spacing). In some specific zones where the geological knowledge was sufficient to do so, up to 135m was used downplunge of the oreshoots to classify as Inferred.

Indicated Mineral Resources were defined where the following criteria were met:

- > Drill spacing of 25 m or less
- > Demonstrated geological continuity
- > Grade continuity

Measured Mineral Resources were defined where the following criteria were met:

- > Drill spacing of 12.5 m or less
- > Demonstrated geological continuity
- > Grade continuity
- > Presence of historical mining (drifts, stopes)

For each individual zone, clipping boundaries were created manually in longitudinal views to either upgrade or downgrade classification in order to homogenize classification by removing artificial features, isolated blocks due to automatically generated classification, and spotted dog effects. All remaining estimated but unclassified blocks were flagged as “Potential” and were not reported. Figure 14-7 shows the classification distribution.

Figure 14-7 : 3D view of the resource classification



14.14 Mining Optimization and Cut-off Grade

Resources were constrained by both economic parameters represented by a cut-off grade and



geometrical parameters represented by pit shells for the open pit resource or stopes shapes for the underground resource. Table 14-11 and Table 14-12 present the economic and geometrical optimization parameters used to constrain the resource.

Table 14-11 : Mineral Resource Estimate Open Pit Optimization parameters

Parameter	Unit	Value				
Selling				Base Case		
Gold price	USD/oz	2,000.00	2,500.00	3,000.00	3,500.00	4,000.00
Exchange Rate	CAD/USD	1.40	1.40	1.40	1.40	1.40
Royalty	%	1.50	1.50	1.50	1.50	1.50
Selling cost	USD/oz	5.00	5.00	5.00	5.00	5.00
Payability	%	99.90	99.90	99.90	99.90	99.90
Net selling price	CAD/oz	2,748.35	3,437.16	4,125.97	4,814.79	5,503.60
Operating costs						
Mining cost	CAD/t mined	6.00	6.00	6.00	6.00	6.00
Process cost	CAD/t milled	25.00	25.00	25.00	25.00	25.00
General & administration cost	CAD/t milled	9.00	9.00	9.00	9.00	10.00
Total mineralized material-based cost	CAD/t milled	34.00	34.00	34.00	34.00	35.00
Processing						
Mill recovery	%	93.1	93.1	93.1	93.1	93.1
Mining						
Block size	m	5x5x5	5x5x5	5x5x5	5x5x5	5x5x5
Slope angle	°	50.0	50.0	50.0	50.0	50.0
Cut-off grade						
Marginal cut-off grade calculated	g/t	0.41	0.33	0.28	0.24	0.21
Marginal cut-off grade rounded	g/t	0.40	0.35	0.30	0.25	0.20



Table 14-12 : Mineral Resource Estimate Underground Optimization parameters

Parameter	Unit	Value				
Selling				Base Case		
Gold price	USD/oz	2,000.00	2,500.00	3,000.00	3,500.00	4,000.00
Exchange Rate	CAD/USD	1.40	1.40	1.40	1.40	1.40
Royalty	%	1.50	1.50	1.50	1.50	1.50
Selling cost	USD/oz	5.00	5.00	5.00	5.00	5.00
Payability	%	99.90	99.90	99.90	99.90	99.90
Net selling price	CAD/oz	2,748.35	3,437.16	4,125.97	4,814.79	5,503.60
Operating costs						
Mining cost	CAD/t mined	90.00	90.00	90.00	90.00	90.00
Process cost	CAD/t milled	25.00	25.00	25.00	25.00	25.00
General & administration cost	CAD/t milled	10.00	10.00	10.00	10.00	10.00
Total mineralized material-based cost	CAD/t milled	125.00	125.00	125.00	125.00	125.00
Processing						
Mill recovery	%	93.10	93.10	93.10	93.10	93.10
Mining						
Minimum mining width Longhole	m	2.50	2.50	2.50	2.50	2.50
Dilution (HW & FW)	m	0.00	0.00	0.00	0.00	0.00
Stope height	m	25.00	25.00	25.00	25.00	25.00
Strike length	m	8.25	8.25	8.25	8.25	8.25
Cut-off grade						
Marginal cut-off grade calculated	g/t	1.52	1.21	1.01	0.87	0.76
Marginal cut-off grade rounded	g/t	1.50	1.20	1.00	0.85	0.75

The open pit Mineral Resource is presented as undiluted and in situ, while the underground Mineral Resource includes internal dilution. The pit optimization to develop the constraining pit shells was done using the pseudoflow algorithm in Deswik.CAD software. The stope optimization to develop the constraining underground stope shapes was done using Deswik.SO software. Revenue calculations were done on a block-by-block basis.

14.15 Mineral Resource Estimate

The 2025 MRE is constrained within a pit shell developed from the above-mentioned pit optimization and DSO shapes using appropriate cut-off grades. Table 14-13 presents the results of the MRE.



Table 14-13 : Cadillac Project Mineral Resource Estimate

	Open Pit Resources			Underground Resources			Total		
	Cut-off Grade = 0.30g/t Au			Cut-off Grade = 1.00g/t Au					
	Tonnes (t)	Grade (Au g/t)	Gold (oz)	Tonnes (t)	Grade (Au g/t)	Gold (oz)	Tonnes (t)	Grade (Au g/t)	Gold (oz)
Measured	1,770,000	2.16	123,300	4,210,000	2.80	379,300	5,988,000	2.61	502,600
Indicated	1,730,000	1.52	84,500	2,240,000	2.51	180,600	3,965,000	2.08	265,200
M&I	3,500,000	1.84	207,800	6,450,000	2.70	559,900	9,953,000	2.40	767,800
Inferred	4,740,000	1.13	172,600	30,450,000	2.29	2,244,200	35,185,000	2.14	2,416,900

Notes to Table 14-13:

- The independent qualified person for the MRE, as defined by National Instrument (“NI”) 43-101 guidelines, is Pierre Luc Richard, P.Geo., of PLR Resources Inc. with contributions from Stephen Coates, P.Eng., of Evomine for cut-off values, open pit optimization solids and underground optimization solids, and Florent Baril, P.Eng., from Bumigeme Inc., for metallurgical parameters. The effective date of the MRE is November 11, 2025.
- These Mineral Resources are not mineral reserves as they have no demonstrated economic viability. No economic evaluation of these Mineral Resource has been produced. The quantity and grade of reported Inferred Resources in this MRE are uncertain in nature and there has been insufficient drilling to define these Inferred Resources as Indicated. However, it is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated category with continued drilling.
- The Qualified Persons are not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, marketing or other relevant issues that could materially affect the Mineral Resource Estimate.
- Calculations used metric units (meters (m), tonnes (t), and g/t). Metal contents in the above table are presented in troy ounces (metric tonne x grade / 31.103475). Values were rounded, and any discrepancies in total amounts are due to rounding errors.
- The Mineral Resource estimate follows the November 29, 2019, CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines.
- Resources are presented as undiluted and in situ for the open-pit scenario and include internal dilution for the underground scenario and are considered to have reasonable prospects for economic extraction. The constraining pit shell was developed using overall pit slopes of 50 degrees. The pit optimization to develop the mineral resource-constraining pit shells was done using the pseudoflow algorithm in Deswik software. The stope optimization to develop the underground mineral resource was done using Deswik.SO software.
- The MRE wireframe was prepared using Leapfrog Edge v.2025.1.1 and is based on 4,477 drill holes, totalling 535,360 meters drilled and 167,978 assays. The cut-off date for the drill hole database was February 17, 2025.
- Composites of 1.0 metre were created inside the mineralization domains. High-grade capping was done on the composited assay data. Based on individual statistical study for each zone, composites were capped between 5.0 g/t Au and 110.0 g/t Au for the high-grade zones, and between 1.0 g/t Au and 5.0 g/t Au for the low-grade zones.
- Pit constrained Mineral Resources for the base case are reported at a cut-off grade of 0.30 g/t Au; DSO-constrained Mineral Resources for the base case are reported at a cut-off grade of 1.00 g/t Au and include internal dilution (must-take). The cut-off grades will be re-evaluated in light of future prevailing market conditions and costs.
- Specific gravity values were estimated using data available in the drill hole database. Density values between 2.80 and 2.88 were applied to the model for hard rock and 2.00 for overburden.
- Grade model resource estimation was calculated from drill hole data using an Ordinary Kriging interpolation method in a sub-blocked model using blocks measuring 5 m x 5 m x 5 m in size and sub-blocks down to 0.625m x 0.625m x 0.625m. Ordinary kriging (OK), inverse square distance (ID2), Nearest neighbour (NN) interpolation methods were tested, resulting in no material difference in the Mineral





Resource Estimates.

12. The Measured, Indicated and Inferred Mineral Resource categories are constrained to areas where drill spacing is less than 12.5m, 25m and 100 metres respectively and show reasonable geological and grade continuity. An additional requirement for the Measured category is the close proximity of underground infrastructure. Cookie cutters were used to define categories based on the above parameters. Based on historical mining and geological knowledge of the deposit, drill spacing was increased up to 140m in the shoot direction to define inferred resources for some zones.

Table 14-14 shows the sensitivity of the block model to grade cut-off for the Measured, Indicated and the Inferred underground Mineral Resource Estimate.



The reader is cautioned that the numbers in Table 14-14 should not be misconstrued with a mineral resource statement.

Figure 14-8 shows a 3D view of the mineralized pits and underground solids used for the MRE.

Figure 14-9 shows the block model with gold grades above 0.30g/t.

Table 14-14 : Underground Stope constrained Resources at various cut-off grades

Gold Price Assumption (\$/oz.)	Cut-off Grade (g/t)	Method	Measured			Indicated			Inferred		
			Tonnes (t)	Grade (g/t)	Gold (oz.)	Tonnes (t)	Grade (g/t)	Gold (oz.)	Tonnes (t)	Grade (g/t)	Gold (oz.)
4,000	0.75	Underground	4,010,000	2.64	339,600	1,580,000	2.18	110,700	33,970,000	1.99	2,169,200
	0.20	Open pit	2,600,000	2.02	169,300	3,500,000	1.29	144,900	15,600,000	0.98	491,300
	-	Total	6,607,000	2.40	508,900	5,084,000	1.56	255,600	49,576,000	1.67	2,660,500
3,500	0.85	Underground	4,250,000	2.72	371,500	1,890,000	2.28	138,300	34,080,000	2.12	2,319,900
	0.25	Open pit	2,130,000	2.02	138,500	2,740,000	1.42	124,900	7,490,000	1.05	251,900
	-	Total	6,382,000	2.49	509,900	4,627,000	1.77	263,200	41,571,000	1.92	2,571,800
3,000 (base case)	1.00	Underground	4,210,000	2.80	379,300	2,240,000	2.51	180,600	30,450,000	2.29	2,244,200
	0.3	Open pit	1,770,000	2.16	123,300	1,730,000	1.52	84,500	4,740,000	1.13	172,600
	-	Total	5,988,000	2.61	502,600	3,965,000	2.08	265,200	35,185,000	2.14	2,416,900
2,500	1.20	Underground	4,230,000	2.91	394,900	2,250,000	2.79	201,500	25,940,000	2.50	2,088,000
	0.35	Open pit	1,190,000	2.43	93,300	1,050,000	1.68	56,700	2,810,000	1.28	115,400
	-	Total	5,421,000	2.80	488,200	3,296,000	2.44	258,200	28,746,000	2.38	2,203,400
2,000	1.50	Underground	4,120,000	3.21	425,500	1,940,000	3.27	203,900	19,990,000	2.81	1,807,400
	0.45	Open pit	370,000	2.92	34,800	530,000	1.86	31,600	1,470,000	1.58	74,600
	-	Total	4,494,000	3.19	460,300	2,472,000	2.96	235,500	21,461,000	2.73	1,882,000



Figure 14-8 : 3D view of the modelled pit shells (red) and the underground solids (yellow)

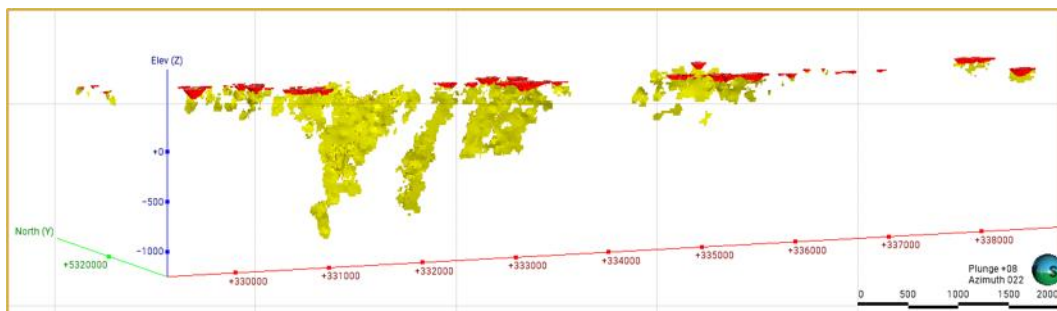
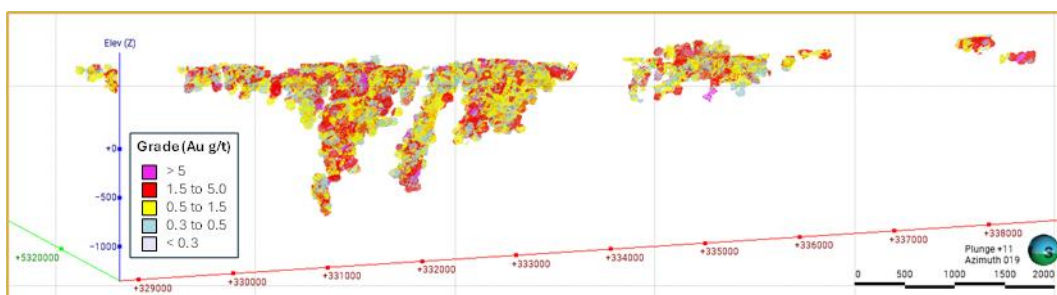


Figure 14-9 : 3D view of the block model with gold grades



15 MINERAL RESERVE ESTIMATE

Mineral Reserves have not been estimated, as the economic analysis is preliminary in nature and includes Inferred Mineral Resources, which are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves.



16 MINING METHODS

16.1 Summary

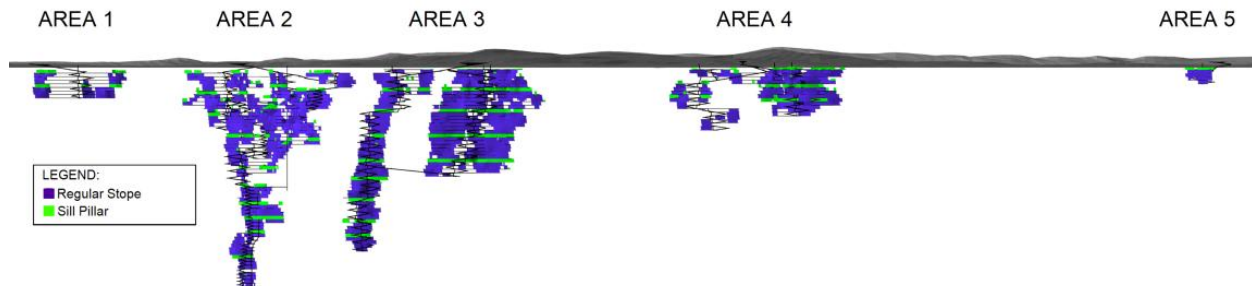
The Cadillac Project is designed as a conventional underground mining operation utilizing longitudinal longhole open stoping with cemented rockfill and pastefill as the primary extraction method. The project schedule begins with a 24-month pre-production period to complete critical underground development. This phase includes constructing ramp access, lateral and vertical levels, ventilation infrastructure, and initial stope preparation. Developing these areas ensures sufficient working faces and operational flexibility to sustain steady-state production upon commissioning.

Following a three-month ramp-up period at the end of pre-production, the mine will reach an initial nominal capacity of 3,000 tonnes per day (tpd), which will be maintained for the first four years of commercial operations. Production will then increase during the fifth year to stabilize at a higher nominal rate of 4,300 tpd for the remainder of the mine life. In total, commercial production will span 16.2 years, bringing the total duration of the project to 18.2 years.

The operation targets six mineralized domains—Main–Chimo, Main–East, Nordeau, Rapid, Contact, and Portal—accessed via five primary surface portals and ramp systems. Each of these five separate mining areas is independently divided into sills, as shown in Figure 16-1, which will be extracted sequentially from the bottom up. Specifically, the divisions are structured as follows: Area 1 (Portal domain) comprises two sills, Area 2 (Main–Chimo domain) comprises 10 sills, Area 3 (Main–East domain) comprises nine sills, Area 4 (Nordeau and Contact domains) comprises three sills, and Area 5 (Rapid domain) comprises one sill.



Figure 16-1 : Mining Areas of the Cadillac Project



Stope dimensions are designed with a 17.5 m strike length and a 25 m vertical height (floor-to-floor sublevel interval). To accommodate equipment constraints and ensure operational practicality, a minimum mining width (MMW) of 2.0 m is applied. Unplanned dilution is accounted for using an Equivalent Linear Overbreak/Slough (ELOS) factor of 0.5 m for the hanging wall and 0.2 m for the footwall; these factors are applied in addition to the MMW. These parameters comply with industry-standard empirical methods and generic rock mass properties typical for this geological setting.

Additionally, stope sub-shapes were optimized to capture marginal mineralized zones. This approach incorporates additional recovered gold ounces from sectors where underground development was already planned, thereby enhancing the project's overall resource utilization.

Mining operations during both pre-production and production will be owner-executed. The primary production fleet includes 10-tonne diesel load-haul-dump (LHD) units and 45-tonne underground haul trucks, complemented by smaller 3.5-tonne LHD units tailored for narrower production drive development, as well as stope mucking and backfilling operations. The haulage truck fleet will transport all mineralized material directly to the surface. Waste rock will be hauled either to the surface waste rock management facility or to designated underground muckbays located near stopes requiring backfill.

Over the life-of-mine (LOM), the underground operation requires approximately 233.8 km of lateral development and 6.4 km of vertical development. A total of 23.1 Mt of mineralized material is scheduled to be mined at an average diluted gold grade of 2.3 g/t.

16.2 Geomechanical and Hydrogeological Characterization

No geomechanical or geotechnical field investigations have been conducted specifically for the purposes of this technical report. The current underground mine design, stope sizing parameters and ground support assumptions rely on historical data and operational records from the past



producing mine, empirical industry standards and operational assumptions from analogous operations in the region, and the experience of the Authors with similar deposits. These parameters also consider the information compiled and documented in the previous Preliminary Economic Assessment (PEA) completed by InnovExplo in 2023.

While these historical data and regional empirical analogies are deemed sufficient for the current level of study, they carry an inherent level of risk regarding long-term ground stability and dilution forecasting. To validate, update, and optimize these historical parameters for the expanded production rates, a comprehensive geotechnical drilling and rock mass characterization program is recommended to support further development of the project. This program is particularly important as several newly targeted mining zones extend well beyond the footprint of the historical operations, where localized rock mass data are currently limited.

16.2.1 Available Geomechanical Data

This section summarizes the available geomechanical information presented in the previous PEA completed by InnovExplo.

A historical technical report prepared by CANMET during the mine's active operating period outlines the following details regarding the original underground operations:

- Zone 5 of the historical Chimo mine is in a sequence of andesites and tuffs located south of their contact with a metasediment formation.
- The regional schistosity is parallel to the stratigraphic units and is oriented N 290 with a dip of 70° N.
- Longitudinal shear zones alternating with schistose and massive amphiboles are typical of the rock mass at Chimo.
- The mineralization is consistent with the stratigraphy as the fault and shear zones contain numerous quartz veins and veinlets with disseminated arsenopyrite in the shales around the quartz veins.

In parallel, CANMET mapped the geological structures present within the levels actively exploited at that time at the Chimo Mine, as shown in Table 16-1.



Table 16-1: Summary of geological structures on the historical Chimo Mine

Level	Zone 5A				Zone 5B			
	Direction (°)	Dip (°)	Spacing (m)	Size (m)	Direction (°)	Dip (°)	Spacing (m)	Size (m)
8	20	90		2 – 3	30	30		
	240	45		3	90	40		
	140	90		3	275	72		
	170	20		2 – 3	50	90		
					335	65		
9	15	80	-5	2	100	80		
	115	80	-5	1	290	70		5 – 10
	335	80	-5	1	20	90		
	32	10	-5	4	160	30		5 – 10
10	60	50	0.2	>3	0	70		
	235	85	0.15	>3	25	75		
	115	15	15	>3	102	65		
					13	15		
11	65	50	2	1				
	160	55	1	3 – 5				
	210	87	15	>3				
	310	40	3	>3				
14	340	88	0	3	30	88	0.5	3 – 5
	30	80	0.6	2.3	210	20	0.6	>5
	290	70	0.5 – 1	>4	115	30	15	1
15	290	75		>5	330	70		2 – 3
	20	75		2 – 5	230	85		3
	30	25		2 – 3				
16	0	75		3	295	77	0.5	>5
	180	25	0.3	5 – 15	0	75	0.5	3
	290	75		>10				
17	40	85	0.1	1	35	80	1	3
	290	80			33	44	1	3
	150	35	1	2 – 3	320	75	1	3
					300	17	1	3 – 5
18	35	70						
	285	75						
	320	75						
	300	35						

16.2.1.1 Rock Mass Classification

CANMET performed some tests to complete the rock mass characterization. The resulting RMR (Bieniawsky, 1976) and Q-system (NGI) parameters for zones 5A and 5B are presented in Table 16-2 and Table 16-3, respectively.



Table 16-2: Rock Mass Rating characterization of the historical Chimo Mine

Parameter	Zone 5A	Zone 5B
Strength of Rock (MPa)	150 – 250	150 – 250
Rating	12	12
RQD (%)	5 – 80	90 – 100
Rating	8 – 17	20
Spacing of joints (m)	0.08 – 0.50	0.30 – 0.70
Rating	12	20
Condition of joints	Roughness varying from moderately rough to very smooth, with little or no altered surface, continuous and sometimes with a slight undulation	Slightly rough surfaces, separation < 1mm, hard joint wall rock
Rating	8	20
Groundwater in joints	Damp	Damp
Rating	10	10
RMR	50 (RQD 50%) – 59 (RQD 80%)	82

Table 16-3: Q-System characterization of the historical Chimo Mine

Parameters	Zone 5A	Zone 5B
RQD (%)	50 – 80	90
Joint Set Number (Jn)	6	6
Joint Roughness Number (Jr)	1.5	2
Joint Alteration Number (Ja)	1.5	0.75
Joint Water Reduction Factor (Jw)	1	1
Stress Reduction Factor (SRF)	5	5
Q Index	1.7 (RQD 50%) – 2.7 (RQD 80%)	8

16.2.2 Hydrogeology

No recent hydrogeological studies have been conducted on the project site. Consequently, the current assessment relies on historical mining data compiled by InnovExplo in the previous PEA from



the 1993 closure and restoration plan of the old Chimo mine.

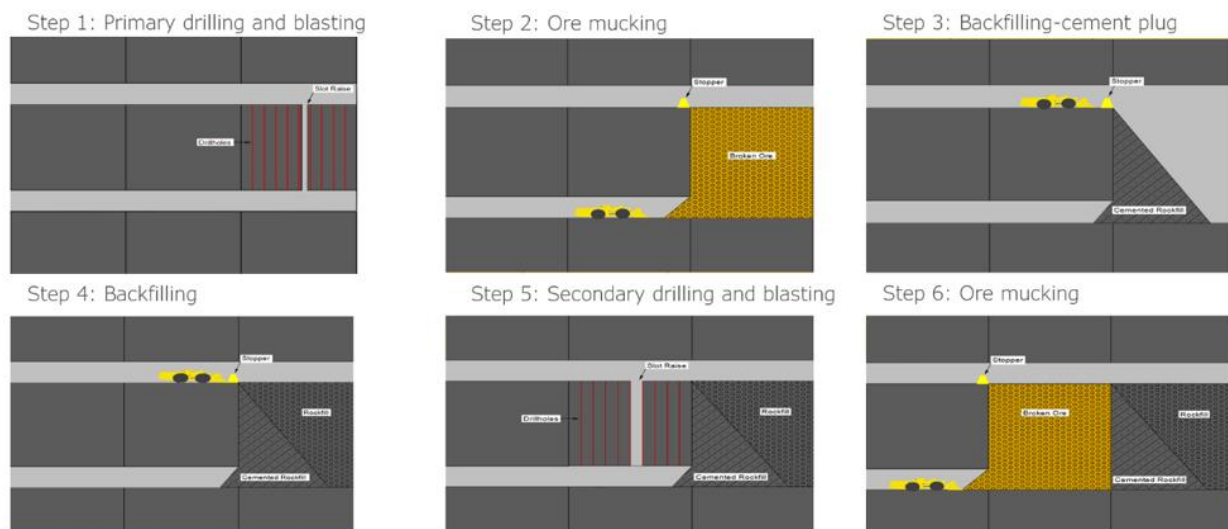
According to InnovExplo's reporting, the summer 1993 hydrogeological study revealed a maximum hydraulic gradient of 0.004 in the groundwater, oriented in a north-west direction perpendicular to the esker. The same study highlighted a hydraulic gradient of 0.00035 within the esker itself, oriented along its north-east axis. Because the permeability of the esker materials (10^{-2} cm/s) is significantly higher than that of the surrounding soils (10^{-5} cm/s), the data suggests that groundwater may partially flow along the esker rather than following the maximum gradient perpendicular to it.

16.3 Underground Mining

16.3.1 Underground Mining Method

The selected underground mining method is longitudinal longhole open stoping (LHOS) with cemented rockfill and pastefill, as illustrated in Figure 16-2. The stoping sequence will ascend from the initial undercut of each mining sill. Generally, stopes will be extracted using an overcut access for drilling and an undercut access for mucking, though the top level of each mining sill will be mined exclusively from the lower access level for both operations. LHOS is a highly productive and cost-effective non-entry method well-suited for competent hard rock deposits, offering high operational flexibility, efficient resource extraction, and enhanced worker safety.

Figure 16-2 : Longhole open stoping mining method



The mining process begins with the development phase, where the ramp, level accesses, and

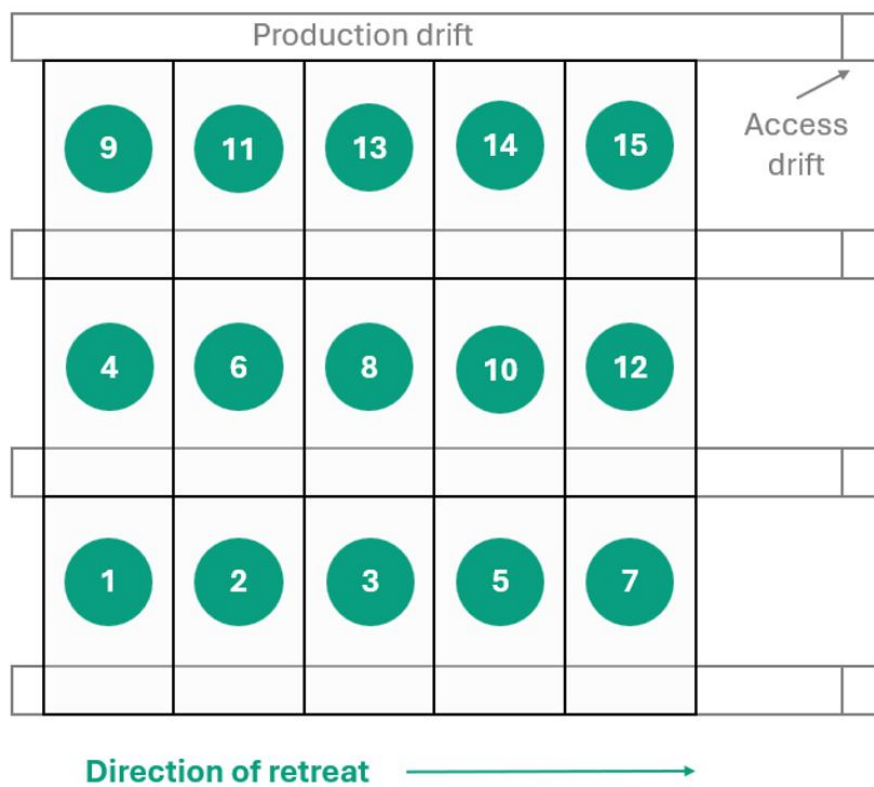
crosscuts are excavated to establish infrastructure and production access. This includes driving overcut drifts for production drilling and undercut drifts for mineralized material extraction along the strike of the mineralization. Once these drifts reach the extremity of the longitudinal mining area, a slot raise, either raise bored or conventionally drilled and blasted, is created to provide the initial void required for production blasting.

The production cycle comprises drilling, blasting, and mucking. Production drilling involves drilling long vertical or inclined holes at regular intervals across the stope. These holes are subsequently loaded with bulk explosives, stemmed, and blasted using controlled techniques to fracture the rock into the slot raise void. After clearing blasting gases, a load-haul-dump (LHD) unit removes the fragmented material through the undercut drift. Due to the non-entry nature of the method, remote-controlled LHD units are utilized to safely and completely clear the stope. Broken material is then transferred to haul trucks and transported to the surface, with multiple blasting and mucking cycles performed depending on stope geometry.

The final phase of the operation is backfilling to maintain regional stability. Depending on the mining sequence, stopes will be filled with cemented material (such as paste fill, hydraulic fill, or cemented rockfill), uncemented rockfill, or a combination of both. Structurally, the longitudinal variant of LHOS is preferred for stoping thicknesses narrower than 8 m. While a transverse variant is typically applied to widths exceeding 8 m, it was excluded from this study due to the relatively narrow nature of the mineralized zones. Subsequent stopes will be mined in a retreating sequence toward the main access of the zone, located either at an extremity or near the middle of the mineralization, as illustrated in the typical longitudinal mining sequence in Figure 16-3.



Figure 16-3 : Typical longitudinal stopeing sequence



16.3.2 Underground Mine Design

16.3.2.1 Stope Design

Stope dimensions were determined based on the Authors' experience in operations under similar conditions, while considering the limit condition parameters estimated by CANMET during the historical Chimo Mine operation, as reported by InnovExplo in the previous PEA and in Table 16-4.



Table 16-4: Limit Conditions of the Historical Chimo Mine according to CANMET

Level		Elevation		Exposed Height (m)	Limit Conditions	
					Good Ground (HR=8.0)	Graphic Schist (HR=5.2)
From	To	From	To		Slope Width (m)	Slope Width (m)
1	2	9917	9879	43	25	14
2	3	9879	9824	60	22	13
3	4	9824	9780	49	24	13
4	5	9780	9736	50	24	13
5	6	9736	9688	53	23	13
6	7	9688	9642	51	23	13
7	8	9642	9588	59	22	13
8	9	9588	9545	48	24	13
9	10	9545	9513	37	28	14
10	11	9513	9481	37	28	14
11	12	9481	9447	39	27	14
12	13	9447	9415	37	28	14
13	14	9415	9383	37	28	14
14	15	9383	9351	37	28	14
15	16	9351	9319	37	28	14
16	17	9319	9287	37	28	14
17	18	9287	9210	82	20	12
18	19	9210	9140	75	20	12

As a result of these considerations, and to ensure the mechanization of the operation in accordance with current industry standards, the stopes were designed with a vertical height of 25 m (floor-to-floor sublevel interval) and a strike length of 17.5 m. These dimensions comply with the sizing proposed by Potvin in his stability graph method for long-hole mining.

Based on the empirical method developed by Clark (1998), unplanned dilution is accounted for using an Equivalent Linear Overbreak/Slough (ELOS) factor of 0.5 m for the hanging wall and 0.2 m for the footwall. These specific parameters were selected based on the Authors' experience in similar deposits.

Given that certain mineralized zones are located near historical mine workings, a 30 m buffer was established around these excavations to ensure that no new stopes are designed within potentially unstable ground.



16.3.2.2 Development Design

Each of the five mining areas at the Cadillac Project will be accessed via a single decline originating from a dedicated portal. Production levels will be accessed by a level access leading to production drives that are in line with the strike of the orebody for longitudinal stoping. A typical level access includes the following infrastructures: sump, electrical bay, muckbay and fresh air access. Table 16-5 summarizes the development lengths required for all mining areas.

Table 16-5: Summary of main underground development lengths

Development Type	Unit	Value
Ramp	m	47,524
Level access	m	14,290
Muckbay	m	9,710
Production drives	m	143,593
Historic development rehabilitation	m	1,015
Infrastructure crosscut	m	16,041
Safety bay	m	1,654
Total lateral underground development	m	233,826
Vertical underground development	m	6,434

Table 16-6 and Table 16-7 detail the underground development types and their dimensions and general underground development parameters, respectively. Figure 16-4 shows a longitudinal view of the underground mine with the zones identified while Figure 16-5 shows a plan view of a typical underground mine production level.

Table 16-6: Underground development types and dimensions

Development type	Width (m)	Height (m)	Length (m)
Ramp	5.0	5.3	variable
Level access (ramp to muckbay)	5.0	5.3	variable
Level access (muckbay to production drive)	4.0	4.0	variable
Production drive	2.7	3.3	variable
Sump	3.7	4.0	12.5
Muckbay	5.0	5.0	17.5
Electrical substation	4.0	4.0	12.5
Ventilation raise access	4.0	4.0	variable
Pumping station	4.0	4.0	12.5
Refuge	5.0	5.0	20.0
Detonator magazine	6.0	5.0	10.0
Explosives magazine	10.0	5.0	25.0



Table 16-7: Underground development parameters

Item	Measurements / Specifications
Ramp gradient	Nominal +/- 15%
Ramp turning radius (minimum)	15.0 m
Ramp - offset from mineralization (minimum)	55.0 m

Figure 16-4 : Longitudinal view of mine

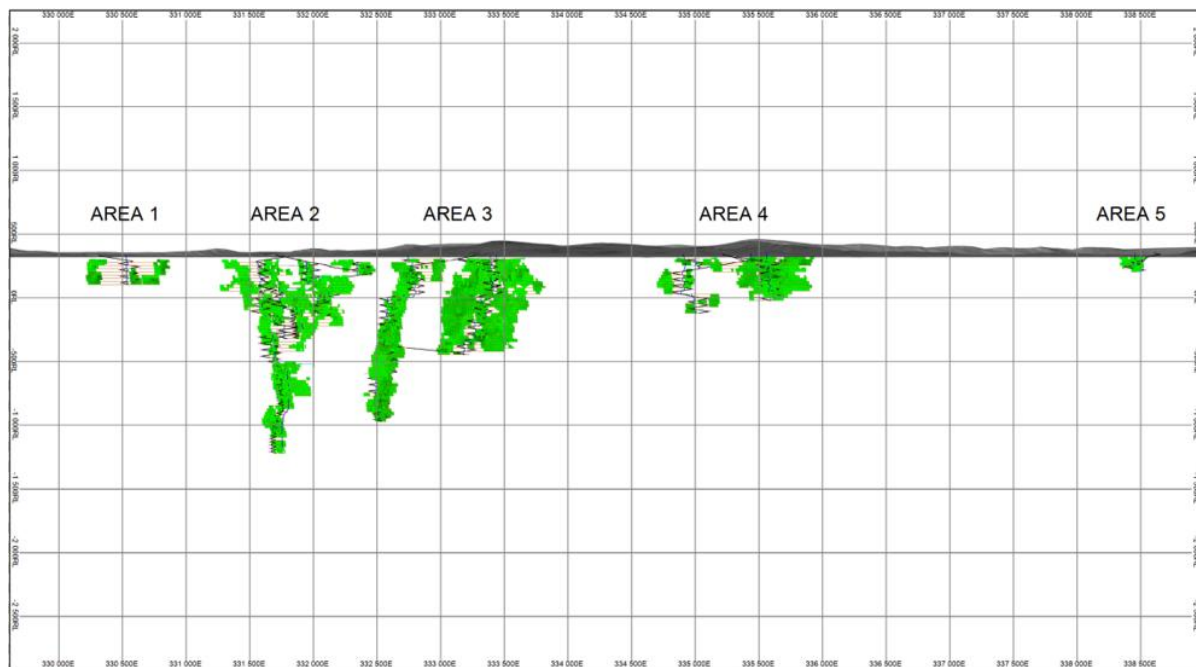
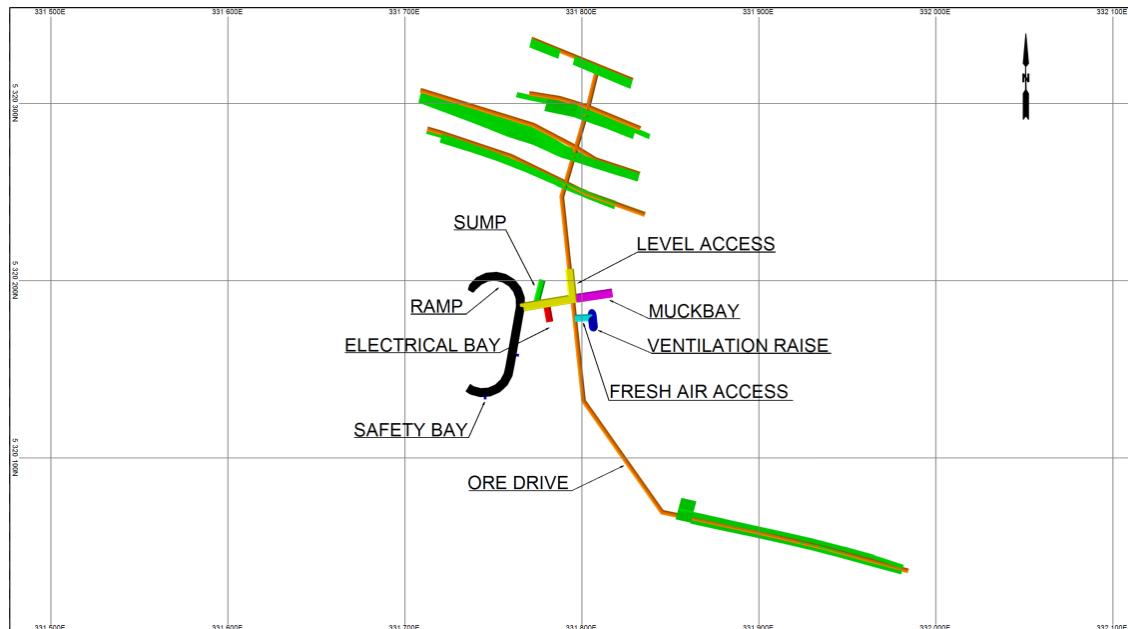


Figure 16-5 : Typical level plan view



16.3.2.3 Sill Pillar Mining

Sill pillars will be temporarily left in place and will serve to increase the number of production centres in concurrent locations. The sill pillars are designed to be 25 m vertical height, like the regular long-hole stope height, and will typically be placed at 100 to 150 m vertical intervals, or the equivalent of four to six mining levels. These sill pillars are planned to be extracted during the mining sequence and will be drilled using uppers.

Since the mining sequence will retreat toward the center of each mineralized zone, causing an increase in induced stresses within the pillars. Considering this and that they will be drilled with uppers, the recovery factor for the sill pillars and the remaining upper stopes is limited to 80%, based on the Authors' experience.

16.3.2.4 Crown Pillar

No formal crown pillar stability assessment was conducted for the scope of this study. The mine design incorporates a 30 m crown pillar, which the Authors consider sufficient to ensure the stability of the stopes and surface infrastructure. This assumption should be validated during subsequent design phases.

16.3.2.5 Backfill

The mining stopes will be backfilled using rock fill, cemented rock fill or pastefill. The integration of



pastefill serves as a major environmental highlight for the project; it allows a significant portion of the mill tailings to be returned underground, effectively reducing the surface tailing storage facility requirements and minimizing the overall surface footprint. Furthermore, this strategy optimizes the material balance by mitigating the need to excavate additional waste rock solely to fulfill backfill requirements.

In further project development phases, it will be required to validate these assumptions by assessing backfill strength requirements. For cost evaluation purposes, 5% cement was considered for most stopes. This percentage is subject to being revised once laboratory tests will have been conducted to ensure that they allow reaching the required strength.

16.3.2.6 Ground Support

The recommended ground support strategies have been developed based on benchmarking data from analogous mining projects, combined with the authors' practical experience in operations sharing similar geomechanical conditions. To validate these assumptions, future geomechanical site investigations and advanced studies will be required during the feasibility stage to refine the ground support parameters and adjust the design requirements accordingly. Moreover, ground support requirements should be reassessed when the rock mass conditions and behaviour are confirmed once underground access becomes available. Changes in ground conditions will need to be monitored and ground support modified accordingly. Table 16-8 summarizes the overall ground support considerations evaluated for this PEA.

Table 16-8: Ground support considerations

Opening	Ground Support considerations
Ramp and other permanent infrastructure such as level accesses	<p>Rebar bolts in the back and in the walls</p> <ul style="list-style-type: none"> > In adverse conditions, a layer of shotcrete is to be added, and swellex bolts may replace the rebar bolts for easier installation. Super swellex can be added as secondary support in the back. This is considered for 10% of the ramp development > For best practice, development faces should be supported with friction bolts and mesh screen
Production drives	Swellex bolts in the back and split sets in the walls, with mesh screen.
Intersections	Secondary support added in the back (resin-grouted threaded bar bolts, or tensioned-plated single-strand plain cablebolts)
Permanent stationary infrastructures (garages, shops)	Rebar bolts in the back and in the walls, with mesh screen, and secondary support in the back (resin-grouted threaded bar bolts, or tensioned-plated single-strand plain cablebolts) with shotcrete
Vertical raises	Rebar bolts in the walls, with mesh screen and possibly a shotcrete layer, depending on conditions



Opening	Ground Support considerations
	<ul style="list-style-type: none"> > Note that inter-level ventilation raises with no man entry can be left unsupported if in good ground. > In unsupported raises where a Laddertube Safescape will be installed, the gap will have to be grouted.
Stopes	Tensioned-plated single-strand plain cablebolts in the back <ul style="list-style-type: none"> > This consideration applies to central and deeper stopes > Depending on the ground conditions and blasting quality, brow reinforcement may be required (rebar bolts)
Surface boxcut excavation	Split sets in the face and in the walls, with mesh screen and shotcrete, and secondary support in the face and in the walls (resin-grouted threaded bar bolts, or tensioned-plated single-strand plain cablebolts)
Mine portal	Rebar bolts with mesh straps and shotcrete in the back and walls in the first 10 m of the ramp. Vertical mesh straps across the brow

16.3.3 Underground Mining Schedule

16.3.3.1 Production Rates

To ensure accurate underground mine scheduling, detailed cycle times were calculated for all primary underground activities using Deswik.Sched™ software and the operational rates shown in Table 16-9. Based on this scheduling model, the mine will achieve an initial nominal capacity of 3,000 tonnes per day (tpd), which will be maintained for the first four years of commercial operations. In the fifth year, production will ramp up and stabilize at a higher nominal rate of 4,300 tpd for the remainder LOM.

Table 16-9: Underground mine scheduling production rates

Parameter	Units	Value
Single Face Development Rate	m/d/unit	4.0
Multi Face Development Rate	m/d/unit	10.0
Stope Preparation	days	3
Slot Raise Drilling Rate	m/d	10
Production Drilling Rate	m/d	250
Production Drilling Factor - Longitudinal	t/m drilled	9.0
Blasting Delay	d	2.5
Mucking Rate	t/d	1,500
Rockfill Rate	t/d	1,200
Cemented Rockfill Rate	t/d	1,000
Cure Time	d	14
Maximum Stoping	t/d	2,000



To consistently maintain these target production rates, multiple mining sills will be operated simultaneously. While these nominal targets are established, actual daily production rates will vary slightly across the LOM due to fluctuations in stope availability and the volume of development mineralized material mined.

16.3.3.2 Overall Mining Schedule

Following the completion of portal excavation and construction in Area 2 (Main-Chimo), underground development will commence with the advance of the main decline toward the high-grade sector. At this early stage, establishing the primary ventilation and emergency egress networks will be prioritized, as these infrastructures are critical to initiating production. Initial mine development will focus on Area 2 (Main-Chimo), with the primary objective of ramping up production in this high-grade zone as quickly as possible. As mining operations in Area 2 (Main-Chimo) advance deeper, Area 3 (Main-East) will be developed via its own dedicated portal. Toward the end of the mine life for Area 3 (Main-East), development of Area 4 (Nordeau and Contact) will commence, followed shortly by the smaller satellite zones, Areas 1 (Portal) and 5 (Rapid). The development schedule for the underground mine by zone is presented in Figure 16-6 while the underground mining schedule is illustrated by material type in Figure 16-7.



Figure 16-6 : Underground mined tonnage by zone according to the Mine Schedule

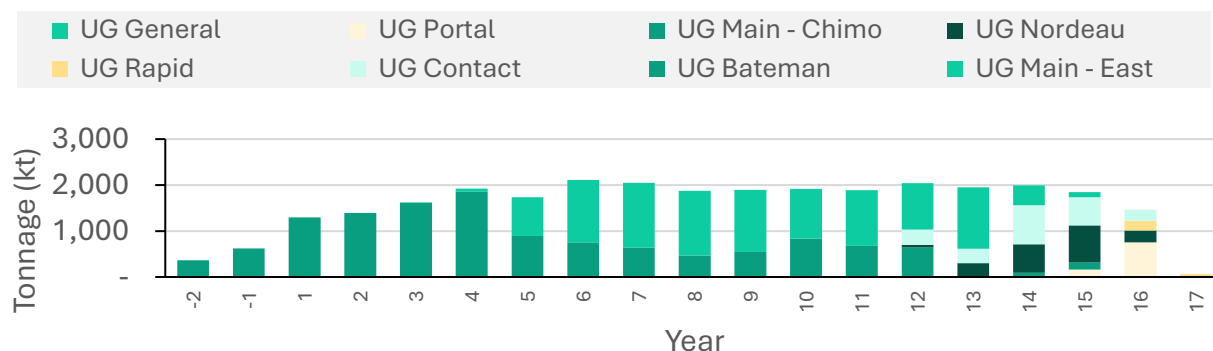


Figure 16-7 : Underground mining schedule by material type



16.3.4 Underground Mining Equipment

The requirements in terms of underground equipment were determined based on the number of operating hours needed to achieve the projected production and development rates outlined in the mine plan. During the production years, haulage cycles consider the distances from the muckbay in the level access drift to the ramp, followed by the ramp ascent to surface. The variable distance between stopes and the muckbays used for loading trucks or picking up waste rock for backfilling is accounted for in calculating LHD requirements.

The quantities of auxiliary equipment were estimated based on the size of the operation or derived from other equipment requirements. Table 16-10 shows the results for the maximum quantity of each equipment type required over the course of the LOM.



Table 16-10: Underground mine mobile equipment fleet

Equipment Type	Maximum Quantity
Definition drill	2
Primary opening drill	2
Production drill	3
Explosives truck	2
LHD – 10 tonnes	2
LHD – 3.5 tonnes	16
Truck	16
Development drill	5
Rockbolter	5
Cablebolter	1
Scissorlift	4
Boom truck	1
Pickup	7
Personnel carriers	2
Cassette carrier	1
Fuel and lube truck	1
Mobile cement batch plant	1
Mechanical service truck	1
Grader	1

16.3.5 Underground Mining Ventilation

Ventilation requirements for the underground mine are primarily based on diesel emissions from the equipment fleet. The minimum ventilation standard of 0.06 m³/kW was used to determine the required airflow per equipment considering some attenuation factors applied based on the estimated equipment utilization. Table 16-11 illustrates the typical ventilation fresh air requirements per equipment used underground.



Table 16-11: Underground mine fresh air requirements per equipment

Equipment	CFM/equipment	Utilization factor	Total quantity	CFM Requirement
Definition drill	11,000	40%	2	8,800
Primary opening drill	11,000	40%	2	8,800
Production drill	11,000	40%	3	13,200
Explosives truck	7,900	60%	2	9,480
LHD – 10 tonnes	20,900	80%	2	33,440
LHD – 3.5 tonnes	10,450	80%	16	133,760
Truck	36,700	80%	16	469,760
Development drill	7,000	40%	5	14,000
Rockbolter	9,200	40%	5	18,400
Cablebolter	9,200	40%	1	3,680
Scissorlift	7,900	60%	4	18,960
Boom truck	7,900	60%	1	4,740
Pickup	10,000	60%	7	42,000
Personnel carriers	10,000	60%	2	12,000
Cassette carrier	10,000	60%	1	6,000
Fuel and lube truck	15,000	40%	1	6,000
Mobile cement batch plant	10,000	40%	1	6,000
Mechanical service truck	15,000	60%	1	9,000
Grader	15,000	60%	1	9,000
Subtotal				827,020
Maximum units in simultaneous operation				721,060
Contingency (10%)				72,106
Total				793,166

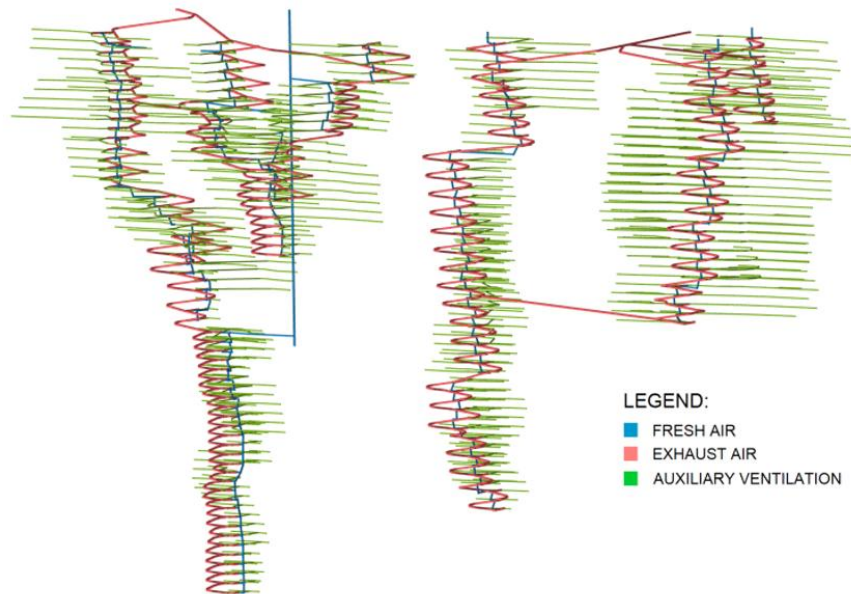
A phased approach will be used to implement the ventilation system. Initially, a temporary system will be put in place to supply fresh air and allow the development of the main decline centered on Area 2 (Main-Chimo) until the permanent ventilation network is developed and commissioned. The temporary ventilation system is designed to provide fresh air requirements for one LHD, two haul trucks and a light vehicle. This entire fresh air volume will be provided by two 48-inch rigid ventilation ducts. Fresh air is supplied to each ventilation line via a containerized ventilation fan system with propane burners for heating during winter months.

The permanent system is designed to accommodate the fresh air requirements of the back end of the pre-production phase and that of full production. The proposed permanent ventilation system is a push ventilation system that consists of a fresh air raise network in each mining area and the respective declines acting as the exhaust. Four ventilation fan and propane burner pairs, two of which will be reused from the temporary system used for initial mine development, are planned for installation in parallel on the surface with all fans pushing air into the mine through the fresh air raise



network. Figure 16-8 illustrates the ventilation network of the underground mine in Area 2 (Main-Chimo) and 3 (Main-East).

Figure 16-8 : Underground mine ventilation network – isometric view



The ventilation system will operate across a range of pressures and flow rates to accommodate the mine's varying operational conditions. Booster fans will be utilized to maintain adequate airflow at depth while keeping overall system pressure within a manageable operational range.

Additionally, the installation of ventilation regulators at the fresh air raise (FAR) access points on each level will allow for precise control of the fresh air volume distributed to individual workplaces. The design parameters for the various fans are summarized in Table 16-12.

Table 16-12: Underground Mine Ventilation System Details

Temporary ventilation fans	Unit	Value
Power	hp	2x 250
Pressure	in wg	14.7
Airflow	CFM	130,000
Main ventilation fans	Unit	Value
Power	hp	4x 500
Pressure	in wg	13.9
Airflow	CFM	408,000
Booster fans	Unit	Value
Power	hp	2x 250
Pressure	in wg	6.0
Airflow	CFM	317,000
Booster fans	Unit	Value
Power	hp	2x 250
Pressure	in wg	6.0
Airflow	CFM	283,000

16.3.6 Underground Mining Services

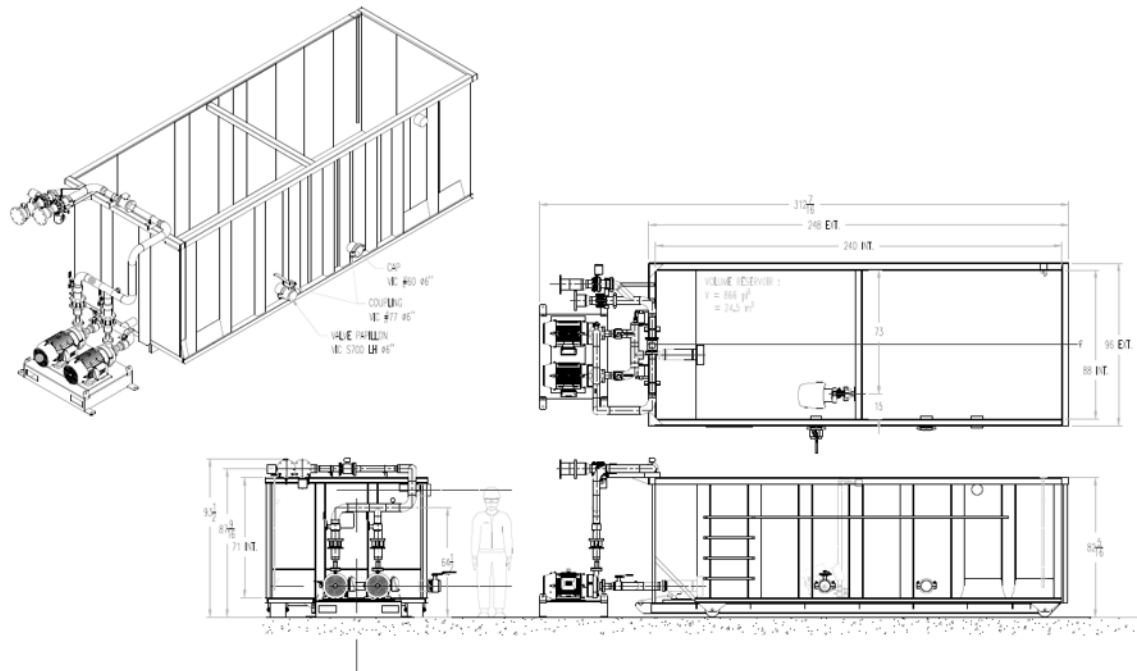
16.3.6.1 Dewatering

The underground mine water management strategy distinguishes between contact water reporting to active underground workings and non-contact groundwater intercepted through depressurization measures. Contact water originating from groundwater inflows into active workings, drilling activities, equipment washing, mine services, and other underground operational activities will be collected at sumps established on each mining level, with the gradient of development on each level directing the contact water towards the sump.

Sumps on consecutive levels will generally be connected via two 4.5-inch drainage holes. At one level per mining sill, this contact water will be transferred with a submersible pump to a staged underground pumping system, which will pump the contact water to surface for storage, management, treatment and release. The planned contact water pumping station arrangement is illustrated in Figure 16-9.



Figure 16-9 : Contact water pumping station arrangement



To reduce groundwater inflows into active workings and minimize hydraulic pressures within major conductive structures, future project development should include studying the possibility of a targeted groundwater depressurization program. This program will utilize strategically oriented dewatering drillholes targeting structures and rock mass intervals exhibiting elevated hydraulic conductivity, as identified through packer testing, structural interpretation, and ongoing hydrogeological assessments. These depressurization drillholes are intended to lower the local water table ahead of mining activities, intercepting groundwater before it reports to underground production areas. Consequently, whenever a new mining sill is brought into production, the groundwater level will have already been drawn down below the base of its planned stopes.

16.3.6.2 Compressed Air

The current PEA considers a compressed air system consisting of 200 HP oil-flooded rotary screw compressors installed within a prefabricated, all-weather building located on surface near the portals. The selected compressors are rated at approximately 1,025 ACFM each at 110 PSIG and feature variable-capacity control systems to optimize operational efficiency under fluctuating demand conditions. The compressed air circuit also includes an air receiver, mist eliminator, automated condensate management system, and associated electrical and control infrastructure. Compressed air will be distributed underground via a 6-inch steel pipeline routed through the mine

services network in the decline and main development headings. This network will supply air-powered equipment—such as dewatering pumps in development headings, handheld drills, and actuators—while also providing an emergency air supply to underground refuge stations. The selected system is designed as a modular, outdoor installation suitable for year-round operation under local climatic conditions, featuring insulated enclosures, heating, ventilation, and heat recovery systems.

16.3.6.3 Communications

The underground communication network consists of an LTE system that will be installed on site and will be expanded over the LOM. Mobile equipment operators, light vehicles, and supervisors will be equipped with LTE phones to communicate with personnel on surface. The LTE system will also enable controlling automation features of some equipment.

16.3.6.4 Fuel Distribution

Fuel will mostly be stored on surface. There will be no fixed underground fuel distribution system. A fuel and lube cassette is planned as part of the mobile equipment fleet to distribute fuel and lube to underground equipment that cannot travel regularly to surface for refueling.

16.3.6.5 Explosives Storage and Handling

Eight underground explosive and detonator magazines will be established at designated locations, with some situated near surface along the main declines and others positioned at strategic levels. Explosives will be delivered to the portal area by the selected supplier, after which they will be transported to the underground magazines via a utility vehicle for secure storage and subsequent distribution.

16.3.6.6 Personnel and Underground Material Transportation

Supplies and personnel will access the underground mine via the main ramps. Light vehicles, including personnel carriers, will be used to transport workers from the surface to the underground mine. Supervisors, technical services, construction and maintenance personnel will also use light vehicles for transportation underground. A flatbed truck equipped with a service boom will be used to move supplies from the surface to the underground active headings, stopes and material storage locations.



16.3.6.7 Equipment Maintenance

Major mechanical maintenance will be performed in the surface workshop. There will be an underground maintenance bay for tires changes and minor repair work. Other small emergency work will be performed underground by mobile maintenance teams.

16.3.7 Underground Mine Safety Measures

16.3.7.1 Emergency Exits

The main decline in each mining area will provide the primary egress from the underground workings, while a dedicated escapeway installed within the primary ventilation raise will serve as the secondary egress to surface.

16.3.7.2 Refuge Stations and Emergency Equipment

Refuge stations will be strategically positioned to ensure that all underground personnel can access a station within 10 minutes of leaving their workplace, or within a maximum travel distance of 1,000 metres. A total of 40 refuge stations is planned to be distributed across the five mining areas. Each refuge station will be equipped with the following:

- Telephone or radio for communication with surface, independent of mine power supply
- Compressed air, water lines, and water supply
- Emergency lighting
- Hand tools and sealing material
- Plan of the underground work showing all exits and the ventilation plans
- All other necessary items according to the applicable regulation
- Fire protection

Underground mobile vehicles will be equipped with automatic fire suppression systems in accordance with best practice. Fire extinguishers will be provided and maintained in accordance with regulations and best practices at electrical installations, pump stations, gear bays, fuel station, service garages and wherever a fire hazard exists. Every vehicle will carry at least one fire extinguisher of adequate size and proper type.

16.3.7.3 Mine Rescue

A fully trained and equipped mine rescue team will be established in accordance with applicable regulations. Mine rescue equipment, including a dedicated underground emergency vehicle and a foam generator, will be available on site.



The rescue team will be trained for surface and underground emergencies. An emergency response plan will be developed and continuously updated as the mine and regulations evolve.

16.3.7.4 Emergency Stench System

A mine stench gas warning system will be installed in the main surface ventilation system, initially in the temporary system and later in the permanent system. This system is designed to inject a specific dosage of stench gas, calculated based on airflow quantity. This gas with a particular smell would alert the workers of an emergency as soon as they smell the gas. Another mine stench gas warning system will be installed at the mine compressed air system as a second means to alert underground workers in the event of an emergency.

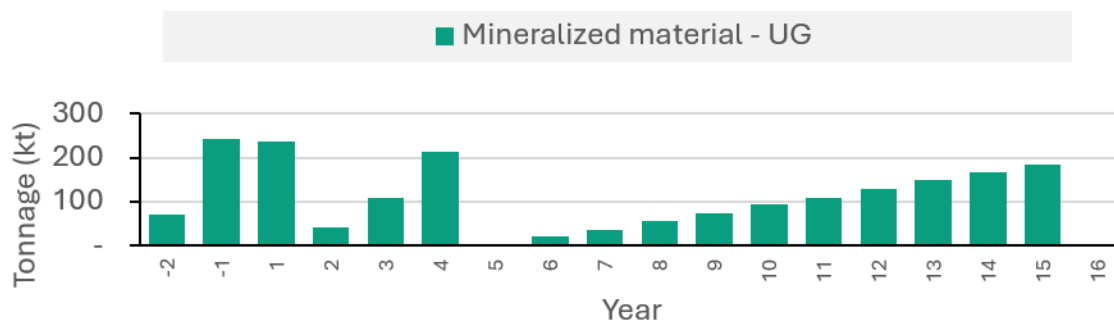
16.4 Surface Mining Stockpiles and Management Facilities

16.4.1 Mineralized Material Stockpiles

Mineralized material stockpiles will be located near each underground mine portal to minimize surface tramming distances for the underground haul trucks. Concrete slabs will be constructed beneath these stockpiles to minimize losses and dilution. During the pre-production phase, a maximum of 240,000 tonnes of mineralized material is anticipated to be stockpiled at the mine site. Upon entering production, stockpile inventory volumes will fluctuate to satisfy mill blending requirements and feed constraints. Figure 16-10 shows the evolution of the closing balance of mineralized material stockpiles over the LOM.



Figure 16-10 : Mineralized material stockpiles closing balance



16.4.2 Waste Rock Management Facilities

A total of 4,411,000 loose cubic metres of waste rock will be mined through underground development over the life of mine, 2,390,000 loose cubic metres of which are planned to be reused in the construction of the tailings management facility (TMF) or as rock fill. This material will be hauled to the waste rock management facilities and stacked using a front-end loader.

A total of five waste rock management facilities (WMF) will be established, each located relatively close to the main portals of the respective declines serving the different mining areas. All facilities are designed with a single lift, 2H:1V bench face slopes, and a 7.5-metre-wide, single-lane access ramp with a 15% gradient. The volumetric capacities of these facilities and the evolution of the total waste volume through the LOM are presented in Figure 16-11, while Figure 16-12 shows the waste volume to be extracted from each zone.

Figure 16-11 : WMF capacities and cumulative waste rock over the LOM

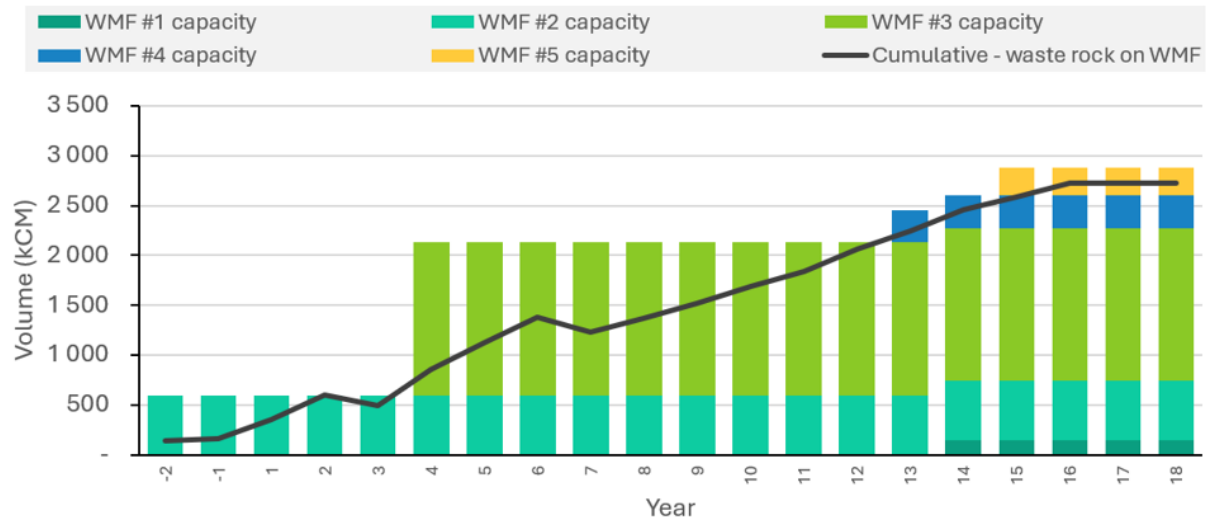
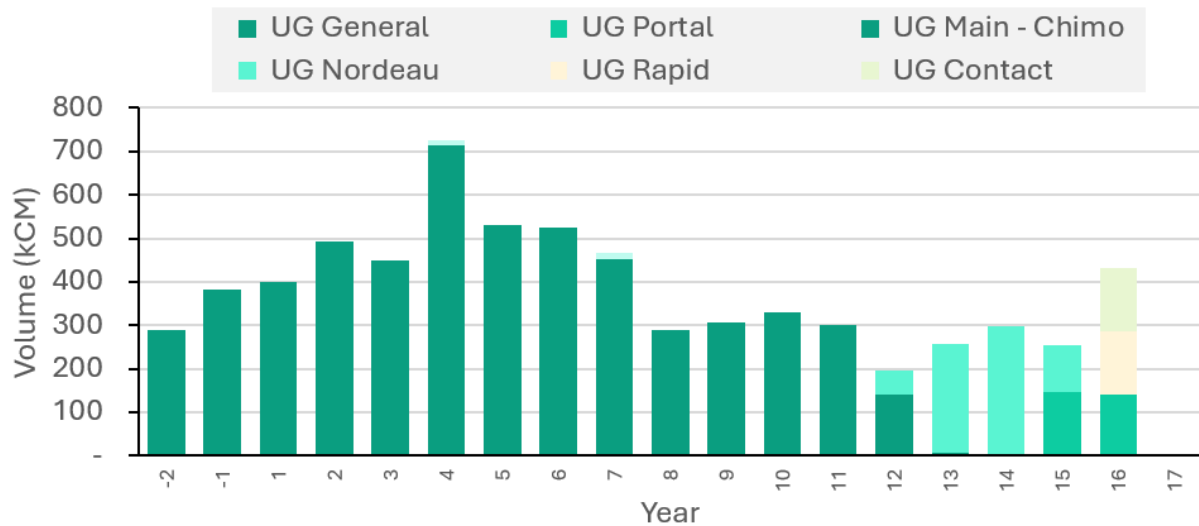


Figure 16-12 : Volume of waste material to be generated by zone



16.4.3 Overburden Management Facilities

A total of approximately 240,000 loose cubic meters of overburden is expected to be excavated mainly from the five mine's portals and from under surface infrastructure. Consequently, a total of five overburden management facilities were designed nearby, with the following capacities: 15, 30, 140, 27, and 28 thousand cubic metres.





All overburden management facilities were designed with a single lift and 3H:1V bench face slopes, and a maximum height of 7.5 meters.

16.5 Overall Mining Schedule

The LOM schedule spans a total of 18.2 years, which includes a two-year pre-production period followed by 16.2 years of commercial operations. Upon completing a three-month ramp-up at the end of pre-production, the plant will operate at an initial nominal rate of 3,000 tpd for Years 1 through 4. Production will scale up during the fifth year, stabilizing at an increased nominal capacity of 4,300 tpd for the remainder of the project life. Table 16-13 summarizes the overall mining schedule.





Table 16-13 : Overall mining schedule summary

	Unit	Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Underground material																					
Mineralized material	kt	23,145	66	240	891	1,082	1,135	1,200	1,200	1,584	1,584	1,584	1,584	1,584	1,584	1,584	1,584	1,584	1,585	1,443	48
	g/t	2.3	3.5	3.0	2.6	2.8	2.8	2.7	2.3	2.0	2.5	2.1	2.0	2.2	2.2	2.4	2.1	1.9	1.9	2.2	3.4
Waste rock	kt	7,198	288	267	368	392	616	752	509	524	451	349	298	454	306	363	349	388	244	279	-
Stockpiles																					
Additions	kt	923	66	186	27	82	80	105	4	26	21	26	26	26	21	25	26	26	22	128	-
Withdrawals	kt	923	-	15	141	95	44	0	252	6	6	6	6	6	6	6	6	6	5	250	67
Closing Balance	kt	-	66	237	123	110	147	252	3	23	38	58	78	97	113	132	151	171	188	67	-
Processing																					
Head feed	kt	23,145	-	69	1,005	1,095	1,098	1,095	1,448	1,564	1,569	1,564	1,564	1,564	1,569	1,564	1,564	1,564	1,569	1,564	115
	g/t	2.3	-	3.1	2.7	2.8	2.8	2.7	2.4	2.0	2.5	2.1	2.0	2.2	2.2	2.4	2.1	1.9	1.9	2.2	2.7
	koz	1,698	-	7	86	99	99	95	110	103	124	107	102	113	110	120	108	96	98	111	10

*Figures may not sum exactly due to rounding.



17 RECOVERY METHODS

17.1 Introduction

Based on the metallurgical testwork results, the process plant was designed with conventional unit operations. Throughput will be 3,000 tonnes per day in Phase 1, increasing to 4,300 tonnes per day in Phase 2. First-year mine production is expected to be processed at a custom milling facility to generate early cash flow. Costs for this period include a realistic custom milling fee and transportation, while recovery is expected to be lower because the mill product P80 will be coarser than 50 μm .

17.2 Overall Process Design

Gold at the Cadillac Project is associated with moderate sulphide contents, primarily pyrrhotite and arsenopyrite. Pyrrhotite is more abundant in the Chimo and East Chimo deposits, whereas arsenopyrite predominates at West Nordeau. Despite this, mineralogy is fairly consistent across the three deposits, and no deleterious minerals were identified. At the design grind size, 43.9% to 61.4% of the gold reports to a gravity concentrate, while cyanidation testwork showed that gold recovery is highly sensitive to grind size. Although whole-rock cyanide leaching remains the selected process route, adding a parallel gravity circuit allows the target recovery to be reached more quickly. Because pyrrhotite and arsenopyrite are expected to increase cyanide and oxygen consumption during leaching, the design includes pre-aeration before cyanide addition to oxidize reactive sulphides, along with lead nitrate addition to precipitate soluble sulphide species. Testwork confirmed that this combination improves gold dissolution kinetics and reduces cyanide consumption.

Review of the available testwork indicated low variability among the three deposits but a strong relationship between gold recovery and grind size. On average, recovery declined by approximately 11 percentage points from the finest to the coarsest grind tested. Additional testing showed that, at the design grind size, a gravity circuit accelerates leach kinetics without affecting final recovery. The selected process route is conventional for a North American gold deposit. Comminution will comprise a SAG mill followed by a ball mill in closed circuit with hydrocyclones to maintain the grind required for target recovery. Part of the hydrocyclone underflow will feed a gravity circuit to recover



liberated gold and accelerate leaching. The design leach residence time includes a margin above the laboratory-measured kinetics.

The Table 17-1 shows the general process design criteria and the feed characteristics:

Table 17-1 General Design Criteria

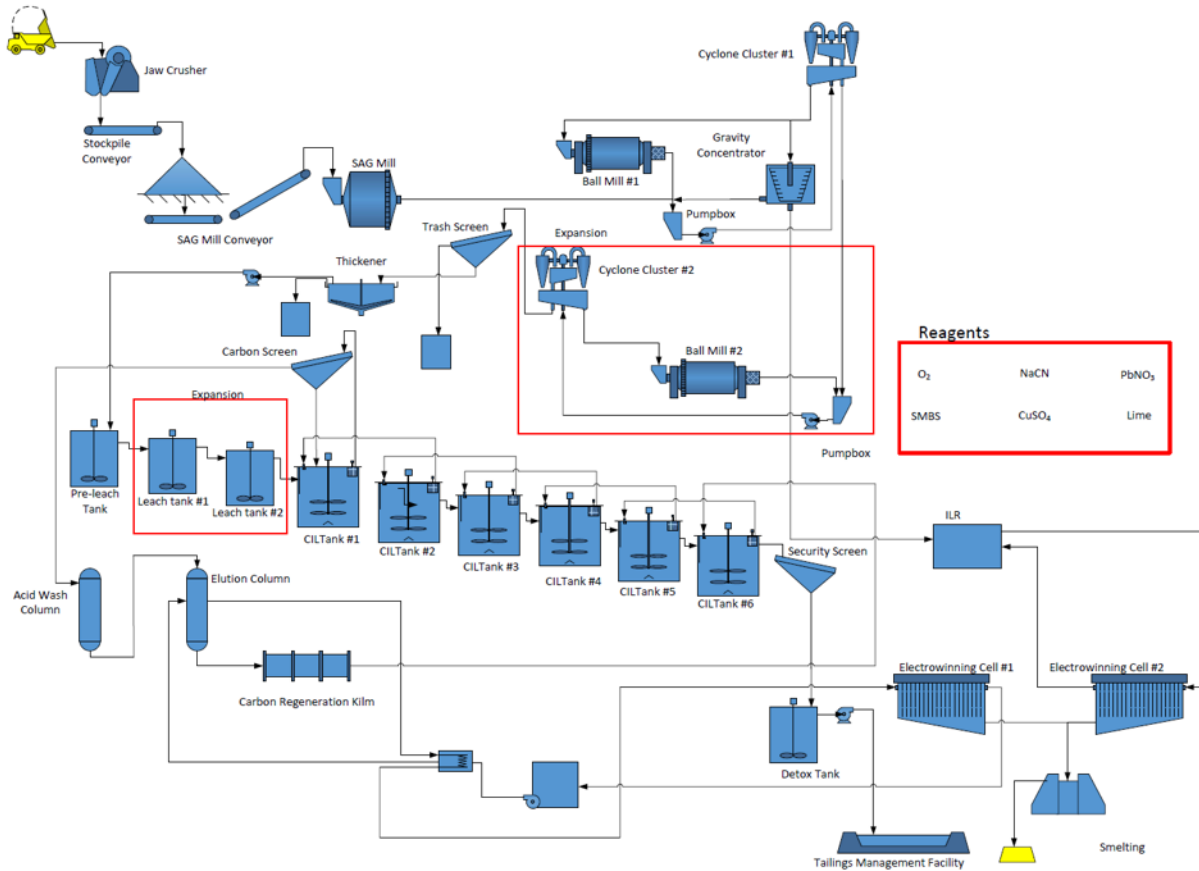
Parameter	Unit	Value Phase 1	Value Phase 2
General mill			
Feed Throughput	t/d	3 000	4 300
Crusher operating	%	65	65
Mill operating	%	92	92
Crusher nominal throughput	t/h	192.3	275.6
Mill nominal throughput	t/h	135.9	194.7
Feed material Characteristic			
Feed Gold Grade	g/t	2.7	2.0
Feed Solid SG	-	2.8	2.8
Chimo - SMC SCSE result	kWh/t	13.1	13.1
East Chimo - SMC SCSE result	kWh/t	11.3	11.3
West Nordeau - SMC SCSE result	kWh/t	11.9	11.9
Chimo - Bond Ball Mill Work Index	kWh/t	13.1	13.1
East Chimo - Bond Ball Mill Work Index	kWh/t	14.3	14.3
West Nordeau - Bond Ball Mill Work Index	kWh/t	13.1	13.1
East Chimo - SMC SCSE result	kWh/t	11.3	11.3
Gravity gold recovery	%	50.0	50.0
Total Gold Recovery	%	95.0	94.8

17.3 Mill Process Plant Description

An overall process flow diagram showing the unit operations in the selected process flowsheet is presented in Figure 17-1.



Figure 17-1 : Overall Process Flow Diagram



The flowsheet reflects the process design criteria established from the metallurgical testwork. It comprises grinding with gravity gold recovery, followed by cyanide leaching and gold recovery using activated carbon. This straightforward flowsheet is widely used in gold processing plants. The following sections describe each process stage.

17.3.1 Primary Crushing & Stockpiling

The crushing circuit is based on the design criteria presented in Table 17-2:

Table 17-2 Crushing Process Design Criteria

Parameter	Unit	Value Phase 1	Value Phase 2
Crushing circuit			
Nominal Hourly Throughput	t/h	192.3	275.6
Design Hourly Throughput	t/h	317.0	317.0
Crusher Opening Dimension	mm	1 250	1 250
Minimum Close Side Setting	mm	100	100
Motor Power	kW	160	160



Parameter	Unit	Value Phase 1	Value Phase 2
Feed F80	mm	700	700
Product P80	mm	150	150
Live stockpile residence time	h	34	24
Live stockpile mass	t	4 300	4 300
Live stockpile proportion	%	25	25
Total stockpile mass	t	17 200	17 200

The crushing circuit is designed to operate 5,694 hours per year at 65% availability. Mine material will be hauled to the jaw crusher, which will process 192 t/h in Phase 1 and 276 t/h in Phase 2. Crushed material will then be conveyed to a 17,200 t stockpile, with 25% live capacity providing 34 hours of storage in Phase 1 and 24 hours in Phase 2. This inventory will allow routine crusher maintenance without interrupting mill feed.

Apron feeders beneath the stockpile will regulate SAG mill feed at 136 t/h in Phase 1 and 195 t/h in Phase 2. Two apron feeders operating in sequence will discharge the material onto the SAG mill feed conveyor.

17.3.2 Grinding Circuit

The grinding circuit is based on the design criteria presented in Table 17-3:

Table 17-3 Grinding Process Design Criteria

Parameter	Unit	Value Phase 1	Value Phase 2
Sag Mill			
Circuit F80	µm	150 000	150 000
Circuit T80	µm	376	1 198
Grinding required power	kW	2 545	2 545
Power efficiency	%	85	85
Motor required motor	kW	2 995	2 995
Motor Installed motor	hp	4 000	4 000
Ball Size	mm	125	125
Ball Consumption	kg/t	0.84	0.59
Ball Mill 1			
Circuit F80	µm	376	1 198
Circuit P80	µm	50	165
Grinding required power	kW	1 511	1 512
Power efficiency	%	90	90
Motor required motor	kW	1 679	1 680



Parameter	Unit	Value Phase 1	Value Phase 2
Motor Installed motor	hp	2 500	2 500
Ball Size	mm	50	62
Ball wearing rate	kg/kWh	0	0
Ball Consumption	kg/t	0.50	0.35
Ball Mill 2			
Circuit F80	µm	-	165
Circuit P80	µm	-	50
Grinding required power	kW	-	1 431
Power efficiency	%	-	90
Motor required motor	kW	-	1 591
Motor Installed motor	hp	-	2 500
Ball Size	mm	-	37
Ball Consumption	kg/t	-	0
Thickener			
Thickener Unit Area	m ² /t/d	0.060	0.060
Thickener required diameter	m	16	19
Thickener designed diameter	m	20	20
Underflow percent solid	%	50	50
Flocculant dosage	g/t	20	20

The grinding circuit will comprise a SAG mill followed by a ball mill operating in closed circuit with hydrocyclones. Phase 1 design criteria are a SAG mill feed F80 of 150 mm and a ball mill product P80 of 50 µm. SAG mill slurry will discharge through a trommel; the undersize will flow to the cyclone feed pump box, while the oversize will be discarded.

A portion of the cyclone underflow will feed the ball mill and the remaining cyclone underflow will feed the gravity circuit. The slurry will pass over a gravity screen, with the oversize returning to the grinding circuit and the undersize feeding the gravity concentrator.

Water will be added to the cyclone feed pump box to achieve the density required for classification. Cyclone overflow will flow by gravity through a trash screen to the pre-leach thickener and then to the leaching circuit.

17.3.3 Gravity Circuit

The gravity circuit is based on the design criteria presented in Table 17-4.



Table 17-4 Gravity Process Design Criteria

Parameter	Unit	Value Phase 1	Value Phase 2
Gravity circuit			
Feed throughput	t/h	85	97
Gravity concentrate mass per batch	kg	30	30
Number of batch per day	#	41	41
Daily gravity concentrate mass	t	1.23	1.23
Gravity concentrate gold grade	g/t	3 389	3 591

A portion of the cyclone underflow will be diverted to the gravity circuit to recover liberated gold early in the process. This will limit gold accumulation in the mills and slurry-handling equipment and improve downstream leach kinetics. No recovery credit is assigned to the gravity circuit because cyanidation of the gravity tailings achieved an overall recovery comparable to whole-rock leaching. The diverted stream will pass over a gravity feed screen. The oversize will return to the grinding circuit, while the undersize will feed a batch-operated centrifugal concentrator. Concentrator tailings will discharge to the cyclone feed pump box for reclassification.

Gravity concentrate will be transferred to an intensive leach reactor feed tank. In the batch reactor, gold will dissolve in a high-strength cyanide solution. The pregnant solution will then flow to the gold-room electrowinning circuit, while the leached solids will return to the grinding circuit. Gold-bearing electrowinning sludge will be smelted to produce doré.

17.3.4 Pre-Leach and Leaching Circuits

The gold leach circuit is based on the design criteria presented in Table 17-5.



Table 17-5 Leaching Process Design Criteria

Parameter	Unit	Value Phase 1	Value Phase 2
Leach circuit			
Pre-aeration time	h	6	6
Required leach time	h	36	36
CIL carbon content	g/L	10	10
Carbon advance rate	t/d	4	4
loaded carbon gold grade	g/t	1 046	1 108
last tank dissolved NaCN	ppm	300	300
Feed throughput	m ³ /h	184	264
Pre-aeration tank			
Quantity of tanks	#	1	1
Effective volume	m ³	1 106	1 106
Leach tank			
Quantity of tanks	#	-	2
Effective volume	m ³	-	2 877
CIL tank			
Quantity of tanks	#	6	6
Effective volume	m ³	6 638	6 638

The pre-leach and leaching circuits comprise a pre-aeration tank followed by six CIL tanks. Trash-screen undersize will feed the thickener, which will stabilize the slurry density entering the leach circuit and thereby reduce variability in gold dissolution and adsorption. This thickening stage is the first step in preparing the slurry for gold leaching. The target thickener underflow density is 50% solids by weight.

The second preparation step occurs in the pre-aeration tank, where the slurry will be oxygenated for six hours. Given the mineralogy of the plant feed, testwork indicated that pre-aeration improves leaching by oxidizing reactive minerals.

In Phase 1, pre-aeration tank discharge will feed the first CIL tank. Oxygen will be sparged into each tank to maintain adequate dissolved oxygen, while hydrated lime and cyanide will be added as required. Lime will maintain a sufficiently high pH to retain cyanide as ions in the slurry and prevent the formation of toxic HCN gas in the work area. Cyanide solution will be introduced into the first tanks to initiate leaching. Fresh and regenerated carbon will enter the last CIL tank and move counter-current to the slurry, advancing from tank to tank toward the first CIL tank using dedicated carbon-transfer pumps. As the carbon advances, it will adsorb dissolved gold. Slurry from the final tank will flow to the detoxification tank for cyanide neutralization.



Interstage screens in each CIL tank will retain the carbon while allowing the slurry to flow by gravity to the next tank. This counter-current sequence will continue until the loaded carbon reaches the first CIL tank. Recessed-impeller pumps will transfer slurry between the CIL tanks and from the first tank to the loaded-carbon screens above the elution circuit.

17.3.5 Cyanide Detoxification and Tailings

The cyanide detoxification circuit is based on the design criteria presented in Table 17-6.

Table 17-6 Cyanide Destruction Process Design Criteria

Parameter	Unit	Value Phase 1	Value Phase 2
Cyanide destruction			
Cyanide Destruction Tank			
Cyanide Destruction Tank % Solids	% w/w	45	45
Number of Unit	-	1	1
Cyanide Destruction Feed Flow Rate	m ³ /h	171.9	246.4
Cyanide Destruction Tank Required Retention Time	h	1	1
Cyanide Destruction Tank Live Volume - Total	m ³	248.9	248.9
Cyanide Destruction pH Target	-	8.5	8.5
Cyanide Destruction CNWAD Feed - Maximum	mg/L	150	150
Cyanide Destruction CNWAD Target	mg/L	1	1
SO ₂ Dosage	g SO ₂ /g CN-	8	8
Copper Sulfate Dosage	g Cu/t mineralized material	50	50
O ₂ / CNWAD Addition Ratio	kg O ₂ /kg CN-	0.5	0.5
O ₂ / SO ₂ Addition Ratio	kg O ₂ /kg SO ₂	0.5	0.5
kg Ca(OH) ₂ / kg SO ₂	kg/kg	0.33	0.33

Leach circuit discharge will flow to the detoxification circuit, where cyanide must be destroyed before the slurry is deposited in the tailings management facility. Over a residence time of approximately 75 minutes, the CNWAD concentration will be reduced from 300 mg/L to less than 5 mg/L to meet environmental discharge requirements.

Cyanide will be destroyed using the SO₂/O₂ process, with oxygen, lime, copper sulphate, and sodium metabisulfite as reagents. The detoxification tank will include oxygen injection points and an agitator to ensure thorough mixing with the tailings slurry. Treated tailings will then flow to the Tailings Management Facility.



17.3.6 Carbon Acid Wash, Elution and Regeneration Circuit

The carbon elution circuit is based on the design criteria presented in Table 17-7.

Table 17-7 Carbon Treatment Process Design Criteria

Parameter	Unit	Value Phase 1	Value Phase 2
Carbon elution and regeneration			
Acid wash column carbon capacity	t	4	4
Elution vessel carbon capacity	t	4	4
Elution bed volume (BV)	m ³	8	8
Elution solution flow	BV/h	2	2
Elution solution flow	m ³ /h	16	16
Elution duration	h	12	12
Electrowinning cell flow capacity	m ³ /h	16	16
Regeneration Kiln operating temperature	°C	600	600
Regeneration Kiln capacity	t/d	4	4

Before gold stripping, the loaded carbon will be treated with a weak hydrochloric acid solution to remove calcium, magnesium, and other salt deposits. These deposits could reduce elution efficiency, become baked onto the carbon during subsequent steps, and ultimately foul the carbon. Loaded carbon from the loaded carbon recovery screen will flow by gravity into the acid wash column. Entrained water will be drained from the column, which will then be refilled from the bottom up with the hydrochloric acid solution. Once filled with acid, the column will be left to soak; the spent acid will then be rinsed from the carbon and discarded.

The acid-washed carbon will then be transferred hydraulically to the elution column for gold stripping.

The gold stripping (elution) circuit will use the Zadra process. An elution solution of water, caustic soda, and sodium cyanide will be prepared. The solution will be pre-heated and passed through a heat exchanger and a heater until it reaches 140°C. It will then flow through the elution column, where it will remove gold from the carbon. From the column, the solution will pass through the heat exchanger to reduce its temperature to below 100°C before being directed to the electrowinning cells, where the gold will be plated. The solution will then return to the elution circuit for reuse. It will pass through the column at a rate of 2 bed volumes per hour for 9 hours. Cool water will then be injected into the column to cool the carbon.

After the cool-down sequence is complete, the carbon will be transferred hydraulically through a



dewatering screen to the carbon regeneration kiln feed hopper.

17.3.6.1 Carbon Reactivation

Carbon will be reactivated in a regeneration kiln. Dewatered barren carbon from the stripping circuit will be retained in a feed hopper, then transferred by a screw feeder into the reactivation kiln. There, it will be heated at high temperature in an atmosphere of superheated steam to restore the carbon's activity.

Carbon discharged from the kiln will be quenched in water and screened on a carbon sizing screen located on top of the CIL tanks to remove undersize carbon fragments. The undersize fine carbon will flow by gravity to a container, while the carbon screen oversize will be directed to the latest CIL tank in the circuit.

17.3.6.2 Electrowinning and Smelting

Gold will be recovered from the pregnant solution by electrowinning, then smelted into doré bars. The pregnant solution will be pumped through multiple electrowinning cells equipped with stainless steel mesh cathodes. Gold will deposit on the cathodes, while the resulting barren solution will be pumped back to the elution circuit.

Gold-rich sludge will be removed from the steel cathodes in the electrowinning cells using high-pressure spray water and will flow by gravity to the sludge hopper. The sludge will be dewatered in a filter press, then manually transported on a tray to the oven for drying.

The dried sludge will be removed from the oven, combined with fluxes in a flux mixer, and then fed to the smelt furnace. After the entire mixture has been added and allowed enough time to melt fully, the slag will be poured into a conical slag pot. The liquid metal will then be poured into moulds on a mound tray. Once cooled, the doré will be cleaned, weighed, and stamped. The bars will be stored in a vault pending shipment.

Dust collection for smelting will be provided in the gold room

Electrowinning and smelting will occur in a secure, supervised gold room.

17.3.7 Reagent Handling and Storage

The reagent consumption is based on the design criteria presented in Table 17-8.



Table 17-8 Reagent Consumption Design Criteria

Parameter	Unit	Value Phase 1	Value Phase 2
Reagent			
Sodium Cyanide			
consumption	kg/t	0.50	0.50
Annual requirement	t/y	548	785
Quick lime			
consumption	kg/t	1.86	1.86
Annual requirement	t/y	2 038	2 921
Sodium Metabisulfite (SMBS)			
consumption	kg/t	1.62	1.62
Annual requirement	t/y	1 771	1 771
Oxygen			
consumption	kg/t	0.61	0.61
Annual requirement	t/y	671	962
Lead Nitrate (Pb(NO ₃) ₂)			
consumption	kg/t	0.100	0.100
Annual requirement	t/y	110	157
Copper Sulphate Pentahydrate (CuSO ₄ *5H ₂ O)			
consumption	kg/t	0.15	0.15
Annual requirement	t/y	161	230
Flocculant			
consumption	kg/t	0.02	0.02
Annual requirement	t/y	22	31
Activated Carbon			
consumption	g/t	0.050	0.035
Annual requirement	t/y	55	55
Sodium Hydroxyde			
consumption	kg/t	0	0
Annual requirement	t/y	85	85
HydroChloric Acid			
consumption	kg/t	0	0
Annual requirement	t/y	139	139

Compatible reagent mixing and storage systems will be installed in containment areas to prevent contact between incompatible reagents. Storage tanks will have level indicators, instrumentation and alarms to prevent spills during normal operation. The facilities will include appropriate ventilation, fire and safety protection, eyewash stations, and safety data sheet (SDS) stations throughout. Sumps and sump pumps will control spillage.



17.3.7.1 Sodium Cyanide

A concentrated sodium cyanide solution will be delivered to site and pumped into the cyanide storage tank using a transfer pump. Dosing pumps will supply sodium cyanide to the leach circuit, the intensive leach reactor and the elution circuit.

17.3.7.2 Quick Lime

Bulk lime will be delivered by tanker and conveyed to the lime silo. It will then be withdrawn from the silo and fed to the lime slaker. The lime distribution tanks will transfer lime solution from the lime mix storage tank to the leaching circuit and cyanide detoxification

17.3.7.3 Sodium Metabisulphite (SMBS)

SMBS will arrive as solid flakes in bulk bags. Process water will first be added to the agitated SMBS mixing tank. A frame and hoist will lift the bags onto the SMBS bag breaker above the tank, allowing the solid reagent to fall into the water and dissolve to the required concentration. Once mixing is complete, the SMBS transfer pump will move the solution to the SMBS storage tank, and the SMBS dosing pump will deliver it to the cyanide detoxification circuit. An extraction fan above the SMBS mixing tank will remove any SO₂ gas generated during mixing.

17.3.7.4 Oxygen

A supplier will provide the oxygen by installing and operating an on-site oxygen generator together with a cryogenic tank. The installation will have no capital cost, and oxygen charges will be based on actual consumption.

17.3.7.5 Lead Nitrate (Pb(NO₃)₂)

Lead Nitrate, supplied as crystals or prills in bulk bags, will be stored in a dedicated, banded area of the reagent building, segregated from acids and combustible materials. A frame and hoist will place the bags on the bag breaker above the agitated lead nitrate mixing tank, where process water will dissolve the reagent to the required concentration. An extraction fan above the bag breaker and mixing tank will remove dust generated while the reagent is added.

17.3.7.6 Copper Sulphate Pentahydrate (CuSO₄ X 5H₂O)

Copper sulphate pentahydrate will arrive as solid crystals in bulk bags. Process water will first be added to the agitated copper sulphate mixing tank. A frame and hoist will lift the bulk bags,



periodically placing one bag on the copper sulphate bag breaker above the tank. The solid reagent will drop into the water and dissolve to the required dosing concentration. The copper sulphate solution will then be transferred to a distribution tank and the copper sulphate dosing pump will deliver the solution to the cyanide detoxification circuit. An extraction fan above the copper sulphate bag breaker and mixing tank will remove reagent dust generated during reagent addition/mixing.

17.3.7.7 Flocculant

Powdered flocculant will arrive on site in bulk bags and be stored in the warehouse. A single self-contained mixing and dosing system will comprise a flocculant storage hopper, flocculant screw feeder, flocculant preparation water heater, flocculant mixing tank, flocculant transfer pump, flocculant storage tank and flocculant dosing pump. The flocculant hoist will load powdered flocculant into the storage hopper. Dry flocculant will then be transferred pneumatically to the wetting head for contact with water.

A 0.50% w/v flocculant solution will be agitated in the flocculant mixing tank for a pre-set period. After this pre-set time, the flocculant transfer pumps will move the solution to the flocculant storage tank. It will be diluted further immediately before the addition point. The system will supply the pre-leach thickener.

17.3.7.8 Activated Carbon

Activated carbon will arrive as solid granules in bulk bags. As required, fresh carbon will be added to the carbon quench tank or directly to the final CIL tank.

17.3.7.9 Sodium Hydroxide

Sodium hydroxide (caustic soda) solution will be delivered in bulk by road tanker and offloaded into the bulk storage tank. Dosing pumps will automatically supply the reagent to required locations, such as the elution circuit, to meet dosing requirements.

17.3.7.10 Hydrochloric Acid

Hydrochloric acid solution will be delivered in bulk by road tanker and offloaded into the bulk storage tank. The hydrochloric acid dosing pump will supply it to the acid wash circuit.

17.3.8 Process and Instrument Air

Compressors will generate the required high pressure air. This high-pressure air will be dried to meet



both plant air and instrument air demand, then distributed through air receivers located throughout the plant.

17.3.9 Water Supply System

Fresh water will be delivered to a raw water storage tank and used wherever clean water with low dissolved solids and low salt content is required. Overflow from the tailings management facility and the pre-leach thickener will satisfy most process water requirements. Raw water and contact water from the tailings will supply the remaining make-up water. A dedicated gland water pump, supplied from the freshwater tank, will provide water for all slurry pump applications in the plant.

17.4 Major Equipment

From the design criteria and the flowsheet, a list of major process equipment has been prepared and is presented in Table 17-9.

Table 17-9 Major Equipment List

Equipment	Number	Phase	Characteristics
Crushing			
Rock Breaker	1	1	
Vibrating Grizzly Feeder	1	1	300 t/h capacity
Jaw Crusher	1	1	1200 X 870 mm opening with 160 kW motor
Crushed Material Apron Feeder #1	2	1	200 t/h capacity
Grinding			
SAG Mill	1	1	8.5 m Diameter X 2.6 m Length (EGL) with 3000 kW motor
Ball Mill #1	1	1	4.5 m Diameter X 5.8 m Length (EGL) with 1850 kW motor
Cyclones Feed Pump	2	1	500 m ³ /h capacity pump
Cyclone Cluster #1	1	1	500 m ³ /h capacity cyclone cluster with spare unit
Ball Mill #2	1	2	4.5 m Diameter X 5.8 m Length (EGL) with 1850 kW motor
Cyclones Feed Pump	2	2	500 m ³ /h capacity pump
Cyclone Cluster #2	1	2	500 m ³ /h capacity cyclone cluster with spare unit



Equipment	Number	Phase	Characteristics
Trash Screen	1	1	
Gravity			
Gravity Screen	1	1	Vibrating, One Deck Screen, 1.5 m x 2.4 m, Opening: 2 mm x 8,8 mm
Gravity Concentrator	1	1	Semi-Batch Concentrator; Maximum Slurry Capacity 200 m ³ /h
Gravity intensive leach reactor	1	1	Batch leach of 1.5 t/d, one batch per day
Leach and carbon recovery			
Preleach Thickener	1	1	High Rate Thickener - 18 m Diameter
Preleach Thickener Overflow Pump	2	1	Water pump 30 kW motor
Preleach Thickener Underflow Pump	2	1	Slurry pump with variable speed 80 kW motor
Pre-aeration Tank	1	1	1100 m ³ tank
Pre-aeration Tank Agitator	1	1	Agitator for suspension of solid
Leach Tank	2	2	1450 m ³ tank
Leach Tank Agitator	2	2	Agitator for suspension of solid
CIL Tank	6	1	1100 m ³ tank
CIL Tank Agitator	6	1	Agitator for suspension of solid
CIL Carbon Pump	6	1	100 m ³ /h slurry pump with recessed impeller
Interstage Screen	6	1	400 m ³ /h slurry screening capacity
Carbon Safety Screen	1	1	
Crane	1	1	
Carbon elution			
Loaded carbon screen	1	1	Vibrating, One Deck Screen, 0.9 m x 3.7 m, Opening: 0,63 mm x 8,8 mm
Elution system	1	1	4 tonne batch elution system including refinery equipment
Reagent			
Flocculant preparation and distribution	1	1	Mixing system with tanks and distribution pumps
Sodium Cyanide storage and distribution	1	1	Liquid sodium cyanide storage tank and distribution pumps



Equipment	Number	Phase	Characteristics
Caustic soda storage and distribution	1	1	Liquid caustic soda storage tank and distribution pumps
Quick Lime Slaking and distribution	1	1	Lime silo, slaker with storage tanks and distribution pumps
Lead nitrate preparation and distribution	1	1	Mixing system with tanks and distribution pumps
Copper sulfate preparation and distribution	1	1	Mixing system with tanks and distribution pumps
Metabisulfite preparation and distribution	1	1	Mixing system with tanks and distribution pumps
Tailings			
Detoxification Tank	1	1	1100 m ³ tank
Detoxification Tank Agitator	1	1	High intensity agitator
Tailings Pump	2	1	160 kW, 250 m ³ /h capacity
Water			
Fresh Water Pump A	2	1	110 kW water pump
Gland Seal Water Pump A	2	1	30 kW water pump
Process Water Pump A	2	1	75 kW water pump
Recirculation Water Pump A	2	1	45 kW water pump

The identified major equipment was sized according to process design criteria, and it allows to estimate the cost of the mill. Prices from budgetary quote or price obtained in similar project are used and factors are applied to determine the cost of other items.

17.5 Workforce

Based on the size of the process equipment the required workforce required to operate the process plant has been determined. The Table 17-10 presents the number of employees for each operation department.

Table 17-10 Process Plant Workforce Requirement

Department	Employees	
	Phase 1	Phase 2
Management	4	4
Metallurgy	6	6



Department	Employees	
	Phase 1	Phase 2
Operation	30	30
Maintenance	20	22
Electrical Department	10	12
Laboratory	10	10
Tailing Storage Facility (TSF)	8	8
Total	88	92

The number of employees could be used to determine the manpower operating cost.

17.6 Power Requirements

For each major equipment, the electrical power as been determined. A factor has been applied to this value to estimate the remaining power of all other minor equipment. This gives a total power requirement of 10.1 MW for Phase 1 and 12.5 MW for Phase 2. While applying the mill availability, it gives a consumption of 63,600 MWh/y for Phase 1 and 79,468 MWh/y for Phase 2.



18 PROJECT INFRASTRUCTURE

18.1 Overview

This section provides an overview of the infrastructure required to initiate and sustain mining operations throughout the project life. The infrastructure scope encompasses an on-site mineral processing plant and associated tailings storage facilities, sized to support the planned production ramp-up from an initial 3,000 tpd to a peak capacity of 4,300 tpd. Additionally, the infrastructure includes all surface installations required to support the underground mining operations over the 16.2 years of production, including site access, administration and maintenance buildings, mine services, water and waste management systems, and electrical and fuel distribution networks.

Major infrastructure components include:

- Access road and internal site roads
- Parking area
- Office and dry building
- Mobile equipment maintenance facility
- Portal and mine services
- Ventilation raises with primary ventilation fans
- Cement storage silos for underground backfill
- Waste rock and overburden stockpiles
- Contact water management system
- Contact water basin
- Water treatment plant
- Fuel storage and distribution station
- Electrical distribution and communication systems

18.2 Site Layout

The overall site layout has been developed to optimize operational efficiency, safety, and environmental management, while accommodating the topographic and environmental constraints of the project area. The final selection and positioning of the surface facilities were heavily informed



by a comprehensive literature review completed by Stantec in 2026. This study documented and characterized the potential physical, biological, and social impacts associated with the project's footprint, ensuring the infrastructure arrangement minimizes environmental risk and serves as a foundation for upcoming field inventories.

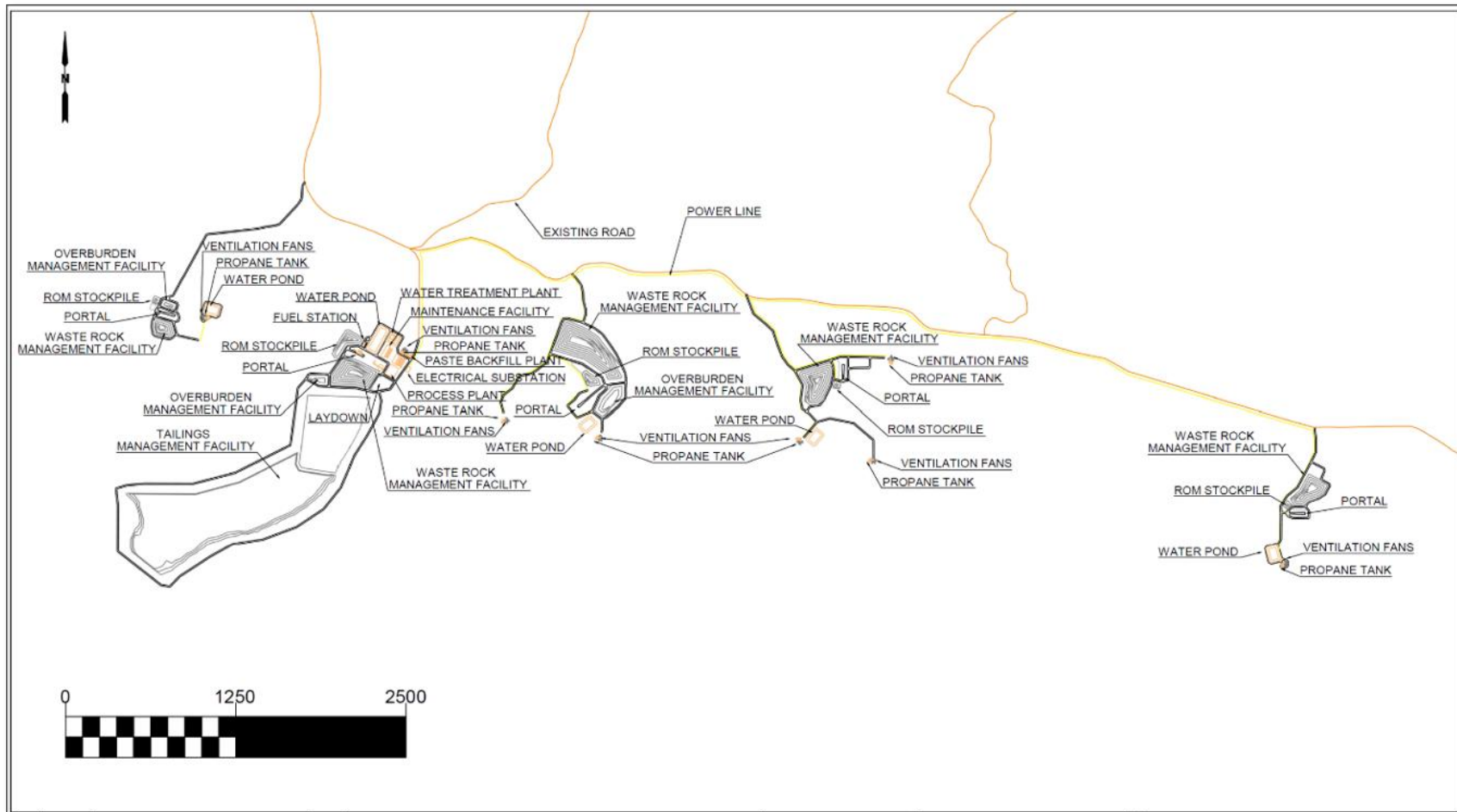
Reflecting these operational and environmental criteria, the surface facilities are organized into distinct functional zones:

- **Mine Access/Portal Area:** The portals and mine support infrastructure are centrally located to minimize haulage distances between underground workings and surface facilities.
- **Administration and Personnel Services:** Administrative and personnel-related facilities are situated near the site entrance to provide controlled access, ensuring clear separation from active mining zones.
- **Maintenance and Support Infrastructure:** Grouped to service the active mining areas efficiently while maintaining a compact footprint.
- **Mineral Processing and Tailings Management:** Integrated into the site footprint to accommodate the on-site processing plant and associated tailings storage facilities.
- **Water Management Infrastructure:** Perimeter drainage ditches surround the operational areas to collect contact water, directing it to a central contact water basin for treatment prior to discharge.
- **Material Storage Areas:** Waste rock, overburden stockpiles, and tailings management facilities are positioned to optimize haulage cycles while maintaining appropriate safety and environmental buffers from active infrastructure and water management systems.

The general arrangement of the surface infrastructure is illustrated in Figure 18-1.



Figure 18-1 : Surface infrastructure general arrangement



18.3 Geotechnical Investigations

To date, no site-specific geotechnical engineering studies or subsurface investigations have been conducted within the project area. Consequently, a comprehensive geotechnical evaluation remains a critical requirement for the future technical studies to ensure the stability and engineering integrity of all major surface infrastructure. Future campaigns must include geotechnical drilling and laboratory testing to characterize soil and bedrock conditions. This data will be essential to perform advanced stability analyses, such as finite element modeling under drained and undrained conditions, required to design foundations for the mineral processing plant and mechanical maintenance buildings, as well as to confirm engineering parameters for the tailings management facilities, waste rock management facilities and overburden management facilities.

Given the absence of site-specific geotechnical investigations, the QP for this section has applied engineering criteria derived from experience with projects under similar conditions to establish a baseline for this Study. Based on these professional assumptions, the proposed basin slopes of 3H:1V and the waste rock management facilities configuration, consisting of 7.5 m bench heights with intermediate berms and an overall slope of approximately 2.5H:1V, are deemed reasonable to meet standard stability criteria at this stage. Similarly, the overburden management facility design utilizing a single lift with 3H:1V slopes is considered acceptable for preliminary planning. For structural foundations, such as the mechanical maintenance building and processing plant, the QPs assume standard bearing capacity constraints typical of the region and expect that a standard compacted granular pad (e.g., MG-80 type) or basic ground preparation will provide sufficient structural support.

All of these geotechnical and foundation design parameters remain preliminary and will be systematically reviewed and optimized once site-specific geotechnical data becomes available for future technical studies.

18.4 Mobile Equipment

Surface mobile equipment will support general site operations, construction activities, and routine infrastructure maintenance. The proposed preliminary fleet consists of a mix of material handling and maintenance vehicles, conceptually including a front-end loader, telehandler, skid steer loader,



forklift, small excavator, motor grader, water truck, pickup trucks, and a dedicated site ambulance.

This equipment will handle material movement, road maintenance, snow removal, site grading, dust suppression, and general logistical support. The initial fleet sizing reflects the conceptual operational requirements of the project and is considered adequate to support routine surface works throughout the life of mine.

Table 18-1: Surface mobile equipment

Equipment type	Quantity
Front-end loader	1
Telehandler	1
Skid steer loader	1
Forklift	1
Small excavator	1
Motor grader	1
Water truck	1
Pickup truck	6
Ambulance	1

18.5 Roads

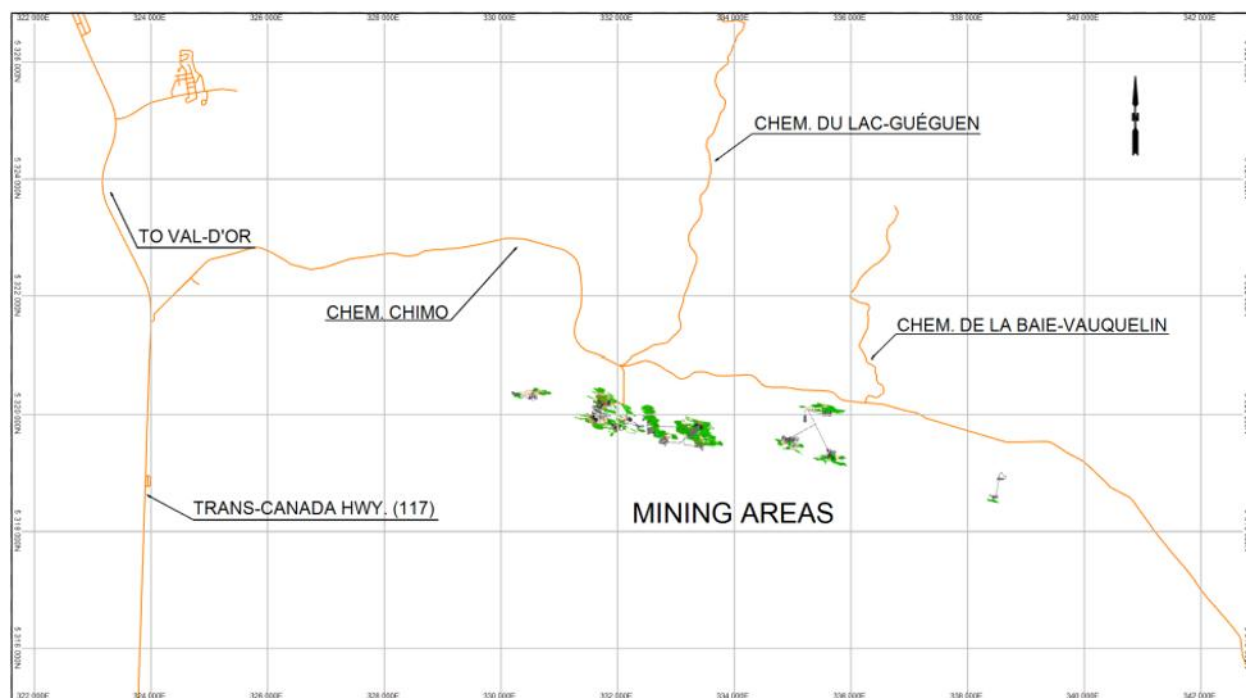
18.5.1 External Roads

The Cadillac Project is located in the Vauquelin Township, approximately 35 km east of the municipality of Val-d'Or, in the Abitibi-Témiscamingue region of Quebec. The site benefits from excellent year-round access via Trans-Canada Highway (Route 117), a major well-developed provincial highway positioned immediately north of the property. Internal site access relies on a network of established gravel roads, including a dedicated logging road that services the historical Portal area and Chimo Mine infrastructure. These transportation infrastructures provide reliable access, ensuring continuous operational capacity throughout the year. Given the industrial scale of planned activities at the Cadillac Project, road refection and maintenance are critical to support the heavy-duty equipment used on site. These roads will be upgraded with additional layers of aggregate and compacted material to increase durability and reduce the maintenance costs associated with the frequent passage of heavy vehicles. As the property's access tracks see some light public usage for local forestry and recreational activities, specific access controls and safety bypass structures are planned to maintain general vehicular flow while ensuring secure and restricted operations around the main mine site areas.



Routine grading and leveling will be integrated into the standard site maintenance schedule to ensure smooth transit and minimize the risk of operational delays due to normal wear or seasonal weather variations. Additionally, existing drainage systems along the property's access roads will be systematically maintained and optimized to prevent localized water accumulation, preserving road surface integrity through regional climate cycles. Aggregates required for ongoing surface maintenance will be sourced efficiently on site, utilizing either local surface materials or suitable waste rock excavated during future underground ramp development. Figure 18-2 illustrates the Cadillac Project's location as well as its direct access from Trans-Canada Highway (Route 117).

Figure 18-2 : Location and access roads



18.5.2 Internal Site Roads

While certain operational haul roads already exist on-site, a network of new internal access roads is proposed to provide site-wide connectivity. These internal roads will connect most of the primary surface infrastructure, including, but not limited to, portal areas, maintenance facility, stockpile areas, process plant, water treatment plant, and paste backfill plant.

The internal roads will be constructed using compacted granular material sourced from an existing local gravel quarry or from crushed waste rock generated during portal excavation.

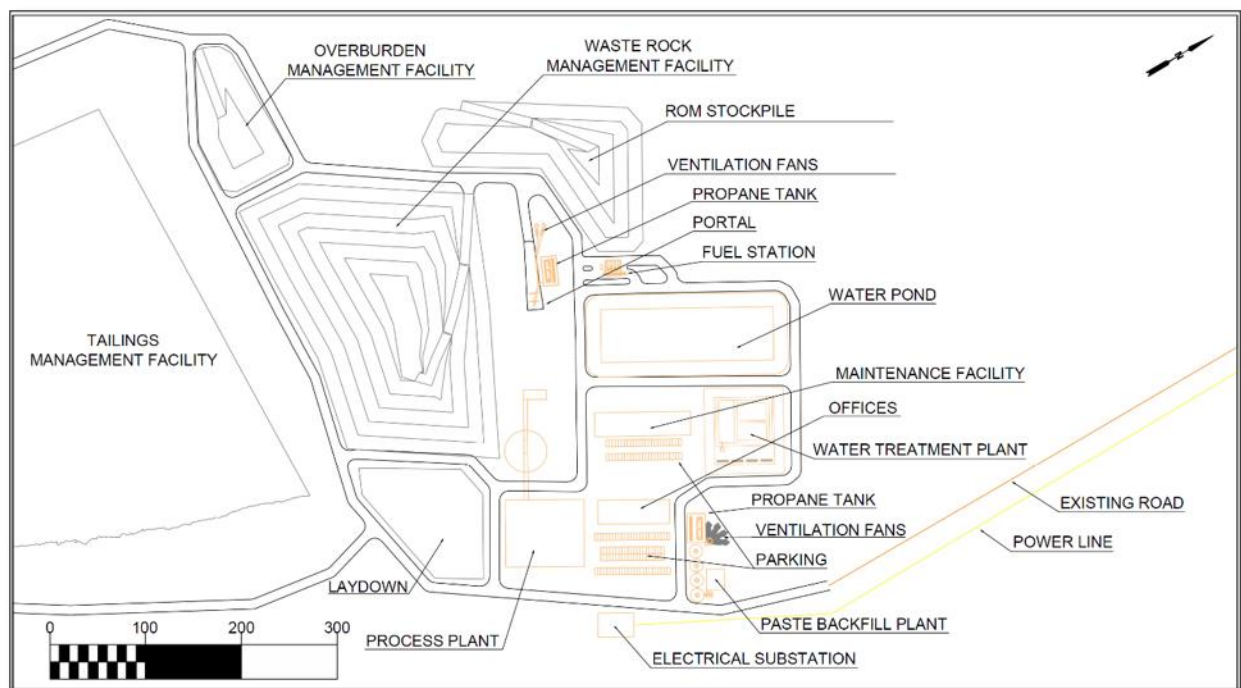


Preliminary road widths will be designed to safely accommodate the anticipated mobile fleet, including heavy mining equipment, support vehicles, and emergency vehicles. Proposed drainage ditches will be integrated along the roadways to manage runoff and maintain structural integrity.

18.6 Site Buildings and Surface Facilities

Most of the surface infrastructure will be centrally located within the Mining Area 2 (Main-Chimo), as shown in Figure 18-3, which serves as the main industrial pad for the entire project.

Figure 18-3 : Main-Chimo industrial pad infrastructure



Infrastructure buildings are conceptually planned as modular, prefabricated structures incorporating steel framing and cladding. At the proposed design level, all facilities will be equipped with smoke, carbon monoxide, and heat detectors, along with appropriate chemical fire extinguishers connected to a central fire alarm panel. The final design and installation of all infrastructure will be engineered to comply with applicable municipal, provincial, and national building codes.

18.6.1 Gatehouse and Guard Facility

An access gatehouse and security facility will be positioned near the project's main entrance to manage and monitor site access. The proposed layout includes a prefabricated, steel-framed

modular control building housing the security and access control office. Security personnel at this location will perform preliminary screenings, ensuring only site-approved and cleared personnel, contractors, and delivery vehicles proceed beyond the access point. Final positioning, building dimensions, and automated access control technologies will be optimized during subsequent engineering phases.

18.6.2 Mine Dry

A modular dry facility will be installed to accommodate both underground and surface workers during shift changes and daily operations. The proposed single-storey structure will feature separate locker areas for clean and dirty clothing, showers, washrooms, change areas, and wash stations. It will also house the mechanical and electrical service rooms required to support operations. Lockers, benches, and drying equipment will be provided for the safe storage of personal protective equipment (PPE) and work clothing, with separate sections for men and women. The facility is intended to connect to the site's water supply, sanitary systems, and electrical grid, and will be sized to provide adequate capacity to support the preliminary workforce requirements.

18.6.3 Office Building

A modular office building will support administrative and operational functions, providing workspace for daily planning and coordination. The facility is preliminarily designed to accommodate permanent personnel, encompassing mine management, technical services, human resources, information technology, accounting, administration, and logistics. The proposed layout features individual offices, open workstations, meeting and conference rooms, a lunchroom, and dedicated mechanical and electrical service rooms. Additionally, it will house an infirmary and a mine rescue room to support emergency preparedness and first-aid response requirements.

18.6.4 Maintenance Facility

A prefabricated structural steel rigid-frame maintenance facility is conceptually designed to support routine servicing, preventive maintenance, and major repairs for the underground mining fleet. The facility footprint is based on preliminary equipment fleet requirements and maintenance planning criteria developed for the Project, ensuring sufficient capacity for the anticipated size and number of mobile equipment units.



The preliminary layout includes service bays, featuring a dedicated wash bay equipped with drainage and water treatment systems to support equipment cleaning and environmental containment. The remaining bays will accommodate maintenance operations, equipment maneuvering, tool storage, and parts warehousing. Final dimensions, layout, and door configurations will be optimized during subsequent engineering phases.

18.6.5 Explosive Storage

A temporary explosives storage facility will be present on surface during the development of the portal box cuts and initial ramp development. It is to be removed once underground explosives storage facilities become available. At that point, explosives arriving on site would be promptly transferred to the underground storage facilities.

18.6.6 Laydown

A surface laydown area is proposed to accommodate bulk materials, consumables, and operational supplies required for underground mining activities. The preliminary footprint incorporates flexible laydown allocations for both indoor and outdoor storage, allowing materials to be distributed across designated zones based on specific operational requirements and protection needs.

The proposed covered storage areas within the laydown are envisioned as prefabricated, lightweight structures that allow rapid construction and minimize civil works requirements relative to conventional buildings. To accommodate varying structural requirements, the preliminary laydown design considers alternative foundations, including both prepared granular and concrete pads. The area also incorporates proposed grading and drainage provisions to prevent water accumulation around the stored materials.

This flexible laydown configuration provides a cost-effective and durable solution for material management under regional climatic conditions, offering protection from precipitation and wind while maintaining operational adaptability. The final laydown dimensions, exact structure types, and specific pad locations will be optimized during subsequent engineering phases as precise supply chain and inventory requirements are finalized.

18.6.7 Fuel Station

Fuel station will be located near the mine portals to store and distribute diesel fuel required for



mobile and stationary equipment. The proposed facility will consist of aboveground diesel storage tanks installed on a concrete foundation, equipped with fuel transfer pumps, dispensing systems, and associated piping to support routine refueling of underground and surface equipment.

The tanks will incorporate a containment system engineered to manage potential spills and comply with applicable environmental and safety regulations. In addition to diesel storage, diesel exhaust fluid will be supplied from mobile or portable tote containers positioned within the refueling area to support emissions-controlled diesel equipment. Final storage capacities, tank configurations, and precise containment dimensions will be optimized during subsequent engineering phases.

18.6.8 Paste Backfill Plant and Cement Storage

A paste backfill plant will be located on surface to support underground mining operations and ground stability requirements. The proposed facility is envisioned to process tailings and other materials to generate paste backfill for distribution underground. The final plant design, processing capacities, equipment selection, and distribution methodologies will be optimized during subsequent engineering phases based on comprehensive metallurgical, rheological, and structural testing.

To support backfilling operations, dry cement will be stored on surface in vertical silos equipped with aeration systems, dust collectors, and access platforms. Bulk cement will be delivered to the site by truck and transferred to the silos via mechanical conveying systems. From the base of the silos, the cement will be pneumatically or gravity-conveyed underground through dedicated borehole infrastructures to supply underground storage systems. The surface installation will include distribution networks, dust collection units, and associated handling equipment to maintain availability for backfill operations. The proposed system configuration and operating concept represent a preliminary baseline that will be finalized as detailed backfill formulations and specifications are matured.

18.6.9 Mine Portals

A total of five mine portals will be developed to provide safe, controlled ramp access to each of the project's distinct mining areas. Each portal layout contemplates a surface box cut involving the excavation of overburden and bedrock to establish a competent, stable portal face. The ramp entrances are preliminarily planned at an approximate -12% gradient, with surface drainage provisions integrated at each entrance to manage runoff and prevent water accumulation within the



underground.

During development, temporary mine services, including compressed air, fresh water, and dewatering pipelines, will be routed along the ramps alongside ventilation ducting to support initial underground operations. Preliminary ground support will be installed within the box cuts and portal collar areas prior to blasting to ensure excavation stability. Waste rock generated during portal development will be transported to designated surface stockpile areas. Final coordinates, exact portal face dimensions, and ground support configurations will be optimized during subsequent engineering phases as localized geotechnical data is finalized.

18.6.10 Mine Ventilation Infrastructure

During early mine development, prior to the commissioning of the permanent primary ventilation raise infrastructures, temporary ventilation systems will be established at each of the five portal areas. These temporary systems are conceptually planned to utilize heating and fan configurations delivering heated fresh air into semi-rigid ventilation ducting extending down each main ramp to support early underground development activities. The ventilation ducts will be supported by proposed structural frame networks mounted on foundations to keep them suspended above the portal entrances, protecting them from ground contact and snow accumulation. These localized temporary systems are intended to provide sufficient airflow and temperature control to support safe ramp development across all mining areas until permanent ventilation becomes operational.

The primary mine ventilation network will rely on surface installations comprising heater and fan systems arranged to deliver heated fresh air into intake raises. Each intake raise will feature a reinforced concrete collar founded directly on competent bedrock following the removal of overburden, which will subsequently be backfilled around the collar to ensure structural stability and surface drainage control. The main ventilation infrastructure will be engineered to supply heated intake air during winter conditions to maintain suitable underground operating temperatures and airflow reliability. In addition, proposed safety egress installations will be constructed at the surface above the mine escapeways, consisting of enclosed structures equipped with airlocks to provide protected emergency access in compliance with applicable mine safety requirements.

18.6.11 Propane Tanks

Propane required for mine ventilation air heating will be supplied from aboveground propane storage tanks installed on prepared foundations. During initial mine development, propane storage will be



strategically positioned near the respective portal areas to supply the temporary heater and fan units used for localized ramp development. Once the primary ventilation raises are commissioned, the propane storage infrastructure will be expanded or relocated adjacent to the permanent ventilation installations to support the primary mine air heating systems.

Associated infrastructure will conceptually include vaporizers, distribution piping, and control equipment required to deliver propane to the ventilation heating units in accordance with applicable environmental, safety, and installation standards. Final tank capacities, total inventory volumes, and precise distribution configurations will be optimized during subsequent engineering phases.

18.6.12 Mine Compressed Air System

Compressed air for underground operations will be supplied by modular, containerized compressor installations housed within fully enclosed, insulated structures. The proposed compressor packages are conceptually planned to feature electric compressors equipped with integrated air drying systems, condensate management networks, ventilation louvers, and interior heating to ensure reliable operation under extreme winter conditions.

The enclosed units will utilize pressure-rated piping and components engineered to deliver compressed air to the underground distribution networks. This containerized configuration is intended to facilitate rapid site installation, reduce environmental exposure, and provide operational reliability suitable for continuous mining operations. The total number of compressor units, specific horsepower ratings, voltage requirements, and geographic distribution across the portal areas will be optimized during subsequent engineering phases to match evolving underground pneumatic demands.

18.6.13 Camp Accommodations

Considering the proximity of the Cadillac Project to Val-d'Or, no on-site mining camp is planned, as it is assumed that the workforce will commute from nearby communities.

18.7 Mineralized Material and Waste Management Facilities

18.7.1 Mineralized Material Stockpiles

A total of five conceptual run-of-mine (ROM) mineralized material stockpiles will be constructed, with one positioned adjacent to each underground mine portal to support the temporary storage and



handling of mineralized material prior to off-site haulage.

These stockpile areas will be integrated with adjacent internal access roads and proposed drainage infrastructure to facilitate safe and efficient site logistics, ensuring continuous material handling and operational adaptability. The preliminary engineering parameters, conceptual storage capacities, operational sequencing, and associated geotechnical criteria for these stockpiles are detailed in Item 16.

18.7.2 Waste Rock Management Facilities

A total of five conceptual waste rock management facilities will be constructed on surface, with one positioned adjacent to each underground mine portal area to facilitate efficient material handling and support underground backfill operations. In this proposed configuration, underground haul trucks that have discharged mineralized material on surface can be reloaded with waste rock at the adjacent waste rock management facility before returning underground. This back-haul process is designed to deliver material for use as backfill in mined-out stopes, improving operational efficiency and minimizing empty haulage.

These decentralized stockpiles will provide the necessary storage capacity and localized operational flexibility required for site-wide waste rock management. All five facilities will be integrated with adjacent internal access roads and proposed drainage infrastructure to ensure safe operations and controlled runoff management. The preliminary engineering parameters, conceptual storage capacities, and associated geotechnical criteria for these waste rock management facilities are detailed in Item 16.

18.7.3 Overburden Management Facilities

Overburden generated during site preparation and construction activities will be managed within five designated overburden management facilities areas, located as shown on the site general arrangement. These stockpiles will be developed on prepared surfaces and integrated with site access roads to allow efficient material placement and retrieval. Drainage controls, including perimeter ditches, will be implemented to manage runoff and direct contact water toward the site water management system. Detailed design, storage volumes, and geotechnical considerations for these facilities are described in Item 16.



18.8 Tailings Management Facility

The Project will require a Tailings Management Facility (TMF) for the containment and long-term management of tailings generated by the mineral processing operations. At the PEA level, the proposed tailings management strategy is based on a conceptual facility configuration developed considering the anticipated tailings production rate, projected mine life, site topography, available land, preliminary geotechnical and environmental information, and overall project infrastructure requirements.

The proposed TMF is located to the south of the industrial pad of the historic Chimo Mine, starting approximately 200m from the process plant. It considers a three-staged construction, with one cell being a raise of the tailings management facility of the historic Chimo Mine. Another cell would be constructed directly to the south, and would also be raised in the course of the life of mine. The facility will be designed to accommodate approximately 9.5 million tonnes of tailings over the anticipated 15.2-year period of on-site processing, which corresponds to the quantity of tailings generated on site that won't be sent underground as paste fill.

The tailings management concept will be further evaluated and refined during subsequent engineering and environmental assessment stages.

18.9 Water Management Facilities

18.9.1 Contact Water

A network of contact water retention basins will be constructed as part of the Project's water management facilities to collect runoff and mine water prior to treatment and discharge. The proposed basins will receive contact water conveyed via gravity through a network of collection ditches surrounding the surface infrastructure, as well as contact water pumped from the underground mining areas.

The basins are conceptually designed to meet applicable regulatory requirements, including the capacity to retain runoff associated with a conservative return-period precipitation event while maintaining appropriate freeboard and incorporating an emergency spillway to safely manage extreme events. The proposed basins will be constructed with an impermeable geomembrane liner system to maintain structural integrity and control environmental containment. The design baseline also includes provisions to manage operational water levels and allow controlled pumping of collected water to the treatment system prior to discharge to the receiving environment.



The final number of basins, precise capacities, engineering dimensions, and localized hydrological design criteria will be optimized during subsequent engineering phases as detailed site-wide water balance modeling is finalized.

18.9.2 Water Treatment Plant

A water treatment plant will be constructed adjacent to the contact water basins to treat collected contact water prior to discharge to the receiving environment. Contact water from site runoff and mine dewatering activities will be conveyed to the water treatment plant following temporary storage within the contact water basin network.

The treatment system will be engineered to meet applicable regulatory discharge criteria prior to the release of treated effluent to the local receiving environment. The facility will conceptually include specialized treatment equipment, pumping systems, chemical dosing systems, and associated piping and instrumentation required to ensure reliable operation and compliance with environmental standards.

The proposed plant location adjacent to the contact water basins is intended to minimize pumping requirements and facilitate efficient transfer of water between storage and treatment facilities, supporting controlled and compliant management of site water throughout the operational period. The final treatment technologies, exact processing capacities, and building footprint will be optimized during subsequent engineering phases as detailed site-wide water balance modeling and effluent toxicity testing are finalized.

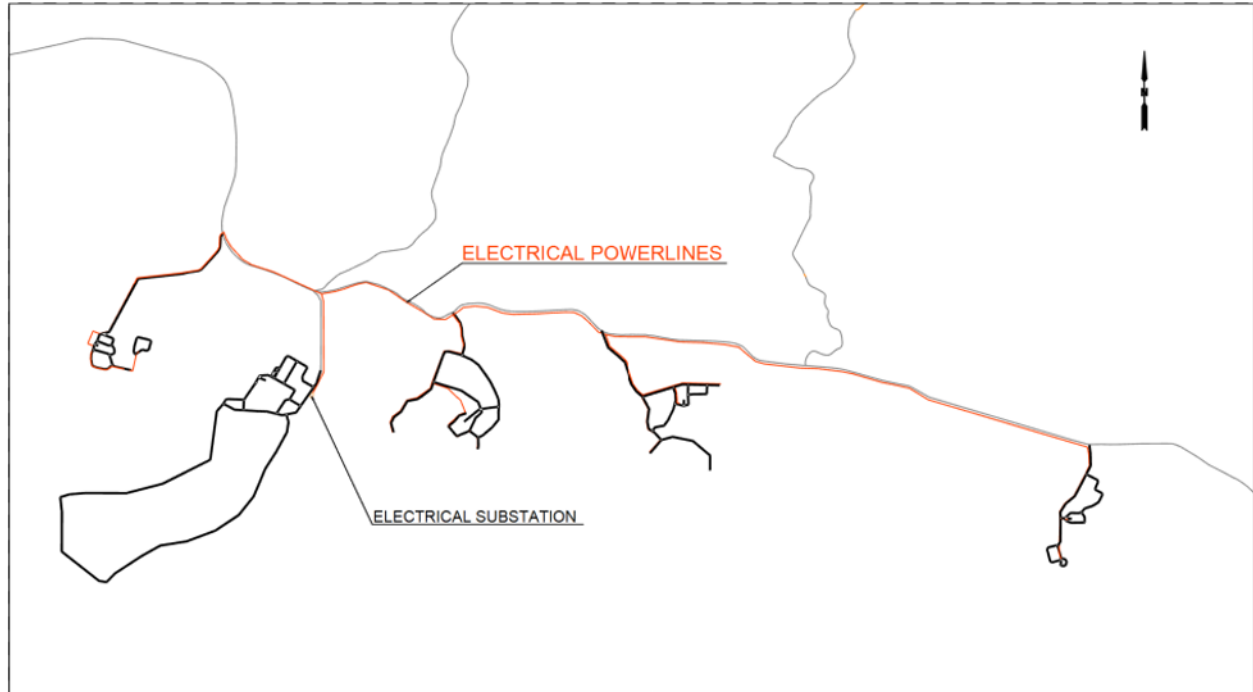
18.10 Electricity and Communications

18.10.1 Electrical Power Transmission Lines

Electrical power for the Cadillac Project is planned to be supplied via the regional electrical grid. The past-producing Chimo Mine property benefits from an existing 25 kV electrical power line infrastructure, which is to be upgraded to 120 kV and stepped down to 13.8 kV at an on-site electrical substation. 13.8 kV electrical power would then be distributed using the proposed site distribution layout shown in Figure 18-4.



Figure 18-4 : Electrical Powerlines of the Cadillac Project



The surface installation is envisioned to utilize pole structures, conductors, insulators, grounding configurations, and associated protection equipment engineered in accordance with Hydro-Québec utility standards and applicable regulatory frameworks. The distribution network will be designed to support the Project's preliminary peak electrical demand.

Civil works are conceptually modeled to include right-of-way clearing, line refurbishment or replacement, pole installation, and access improvements where necessary. The power distribution network will terminate at the proposed mine site main substation, where electrical power will be transformed and distributed to surface facilities and underground loads across the portal areas. This grid connection is intended to provide a reliable power supply, minimizing or eliminating the requirement for large-scale on-site power generation during normal operations. Final transmission capacities, line length extensions, and exact utility interconnection parameters will be optimized during subsequent advanced engineering phases.

18.10.2 Surface Electrical Substation

A main electrical substation will be installed on surface to receive power from the incoming 120 kV line and step it down to 13.8 kV for site-wide distribution. The proposed substation layout will incorporate transformers, circuit breakers, isolation switches, and associated protection and control equipment housed within a dedicated enclosure. This centralized facility is intended to supply reliable electrical power to the process plant, underground mine workings via the portal areas, surface buildings, and other primary infrastructure.

Electrical power received at the mine site will be distributed through a proposed surface power distribution network consisting of step-down transformers, switchgear, and low- and medium-voltage distribution circuits supplying all surface infrastructure. The system will feature localized step-down configurations to supply power at appropriate voltages to major facilities, including the maintenance building, office complex, ventilation systems, containerized compressors, water treatment plant, and fuel handling systems.

Surface distribution will be achieved through a combination of conceptual overhead and buried cabling systems, depending on final location and operational safety requirements. The preliminary electrical network is envisioned to include motor control centers, distribution panels, protective relays, and grounding systems designed to ensure safe and reliable operation. Critical systems will be supported by uninterruptible power supplies or backup auxiliary power systems where required. The surface electrical system will be conceptually engineered to accommodate operational flexibility, staged development, and future expansion.

The final substation footprint, exact transformer capacities, voltage ratings, cable routing, and emergency backup specifications will be optimized during subsequent advanced engineering phases as the project's total connected electrical load is finalized.



18.10.3 Underground Electrical Power Distribution

Electrical power will be distributed underground from the surface substation network via medium-voltage feeders (nominally rated at 13.8 kV) installed along the main ramps and main ventilation raises, supplying electrical power to mining equipment and underground infrastructure across all five mining areas. This power will be transmitted to strategic underground locations, where proposed localized transformers will step down the voltage (to 600 V and 120 V) for use by mobile equipment, fixed installations, and services at the working faces.

The network will supply electrical power to localized ventilation fans, dewatering pump networks, development equipment, lighting systems, communication loops, and auxiliary services. Electrical cables will be routed along each main ramp and within underground workings using protected routing and support systems engineered to withstand dynamic mining conditions. Distribution panels and switchgear will be positioned in designated underground electrical stations to allow safe isolation, control, and maintenance.

The underground electrical network will incorporate appropriate protection systems, grounding, and redundancy to ensure safe and reliable operations. The system configuration is intended to support progressive, staged mine development, allowing for the seamless extension of the electrical infrastructure as the five distinct underground workings advance. The final voltage ratings, exact underground station locations, transformer capacities, and cable distribution routes will be optimized during subsequent advanced engineering phases as localized peak loads are modeled.

18.10.4 Communications

A private LTE communications network is proposed to provide reliable voice, data, and positioning capabilities across all surface facilities and underground operations. The system baseline contemplates a core private network capable of supporting the anticipated workforce requirements, along with associated servers, switching equipment, and network management platforms.

Surface coverage will be provided through radio base stations and directional antennas installed at strategic site locations. Underground communications across the five distinct ramps will be supported by leaky feeder (radiating cable) systems, underground antennas, and localized radio units installed throughout the active mine workings.

The infrastructure will also integrate a Push-to-Talk (PTT) communication system, enabling real-time voice contact between personnel, alongside an underground positioning system to allow tracking of personnel and mobile equipment. The distribution network will rely on fiber-optic backbone cabling,



uninterruptible power supplies (UPS) with battery backup, and associated networking equipment to ensure operational reliability. Engineering, integration, commissioning, and training services will be factored into the deployment to provide a fully integrated communications platform suitable for modern underground mining operations. The final network architecture, specific spectrum/technology generations, total user capacities, and exact infrastructure routing will be optimized during subsequent advanced engineering phases.

18.11 Offsite Infrastructure

The processing strategy for the Cadillac Project contemplates a staged development approach. During the pre-production period and the first year of commercial operations, mineralized material is planned to be transported off-site to a third-party processing facility under a toll milling agreement at a rate of 3,000 tpd.

To support this initial phase, mineralized material will be stockpiled at the third-party mill site until sufficient quantities accumulate for batch processing. Proposed infrastructure to be established at the toll mill site includes a certified weigh station, fencing around the designated stockpile zone, and localized security systems.

The final selection of the third-party mill, precise transportation logistics, on-site plant processing technologies, and expansion footprints will be optimized during subsequent advanced engineering phases.



19 MARKET STUDIES AND CONTRACTS

19.1 Gold Market

Gold is a freely traded global commodity within a well-established, mature, and highly liquid market, often serving as a primary safe-haven asset for investors. It is traded continuously by international banks, financial institutions, and bullion dealers at a spot price for immediate delivery, with standard quotations denominated in US dollars per troy ounce (US\$/oz).

Neither Cartier Resources Inc (“Cartier”) nor its independent consultants have commissioned formal, project-specific market studies for the sale of the gold doré or concentrates from the Cadillac Project. In accordance with NI 43-101 guidelines for Preliminary Economic Assessments (PEA), the Qualified Person (QP) has reviewed prevailing macroeconomic trends, historical price data, and consensus forecasts. Based on this review, the QP has confirmed that the commodity price assumptions adopted for the economic analysis and financial modeling are reasonable, industry-standard, and appropriate.

Global demand for gold remains robust, driven by investment flows, central bank reserves, and industrial or jewelry applications. Given the high liquidity of international gold markets and the extensive refining capacity available both globally and within Canada, the Cadillac Project faces negligible market risk regarding product placement. It is anticipated that the gold doré produced can be readily sold on competitive commercial terms to reputable refiners or smelters.

At present, no material long-term commercial agreements, off-take contracts, hedging arrangements, streaming agreements, or fixed-price sales contracts have been established for the Cadillac Project. This strategy preserves full exposure to prevailing spot market prices at the time of future production while maintaining operational and commercial flexibility.

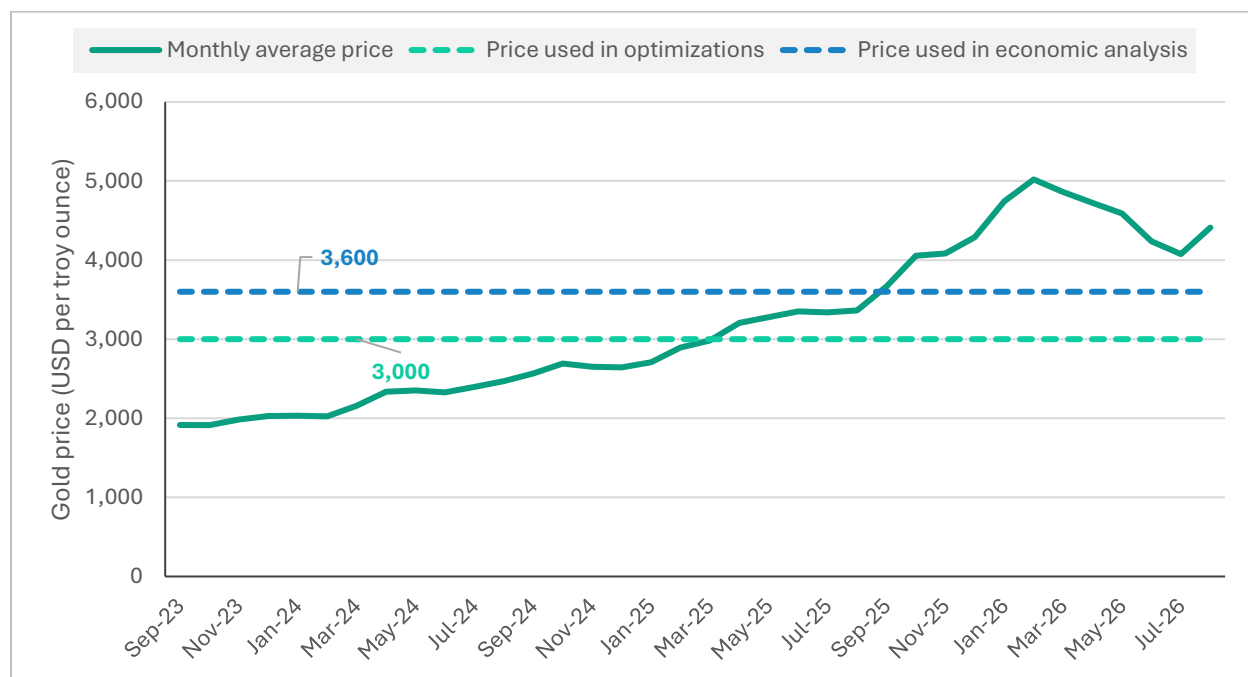
19.2 Metal Price

The price of gold is the primary factor in determining the operational profitability and project cash flows. Consequently, the financial performance of the Cadillac Project is directly linked to the prevailing gold market conditions. On August 31, 2026, the three-year trailing monthly average gold price stood at USD 3,177 per troy ounce, reflecting the commodity's historical trend and recent upward momentum. Figure 19-1 illustrates the historical monthly average values of gold over this three-year period. To reflect current market conditions while maintaining a reasonable economic



buffer, a base case gold price assumption of USD 3,600 per troy ounce has been adopted for the economic analysis and financial modeling of this PEA.

Figure 19-1 : Gold Price

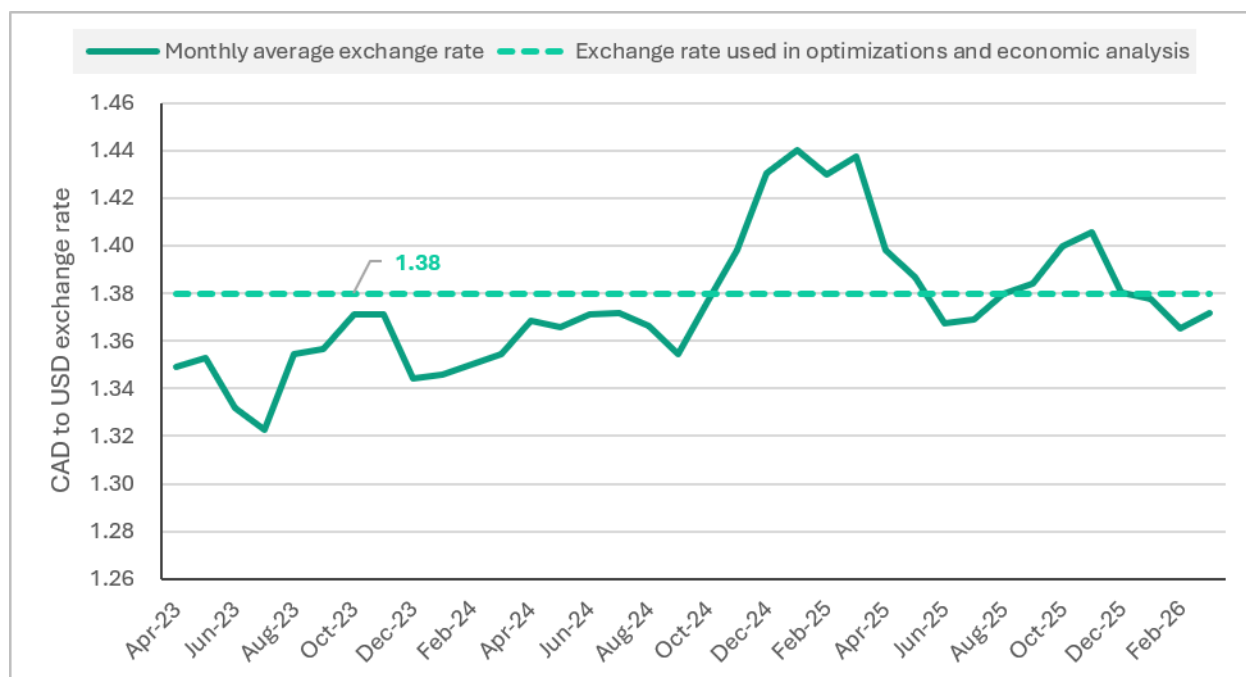


19.3 Exchange Rate

The economic analysis for the Cadillac Project has been prepared using a base case foreign exchange rate of 1.38 Canadian dollars (CAD) per 1.00 United States dollar (USD). This exchange rate has been applied consistently throughout the capital and operating cost estimates, revenue projections, and financial modeling where currency conversions are required. The selected exchange rate is considered reasonable for the purposes of this study, based on recent historical trends and prevailing macroeconomic conditions at the time of preparation of this Preliminary Economic Assessment (PEA). Sensitivity analyses have been conducted to evaluate the impact of variations in foreign exchange rate assumptions on Project economics, as currency fluctuations represent a potential source of financial risk. Because gold sales are denominated in USD while most Project costs are incurred in CAD, a weaker Canadian dollar relative to the United States dollar is generally favorable to the Project economics. Figure 19-2 shows the historical Bank of Canada monthly average exchange rate for the last three years.



Figure 19-2 : CAD to USD Exchange Rate



19.4 Toll Milling

The Project's development strategy contemplates the processing of mineralized material at existing third-party toll milling facilities during the pre-production phase and the first year of commercial production. This transitional approach generates early cash flow prior to the commissioning of the Project's own dedicated processing plant, which is scheduled to come online at the beginning of Year 2.

As of the effective date of this report, Cartier has not received any non-binding Letters of Intent (LOIs) regarding toll milling arrangements. However, the Qualified Persons (QPs) have based the project's processing assumptions on standard, publicly available commercial terms and regional practices for toll milling operations within the Abitibi greenstone belt, where multiple custom milling facilities historically and currently operate.

19.5 Other Contracts

Insurance, transportation and refining contracts for gold doré bars will be negotiated and finalized during the construction phase of the project. Costs for insurance, transportation and refining were estimated to be USD 5 per troy ounce of gold.



20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

This section presents available information on environmental, permitting and social or community requirements related to the Cadillac project. The following sections include available data for environmental studies assessments, waste rocks, tailings and water management, general overview of permit requirements, social and community requirements, and mine closure requirements.

20.1 Environmental Studies

The environmental permitting process requires an understanding of the physical (surface water, groundwater, air, noise, etc.), biological (fauna and flora) and human (land and resource uses, archaeology, landscape, etc.) environments. Early in the permitting process, it is recommended to start baseline studies to identify any fatal flaw or environmental issues (e.g., protected areas, species at risk, waterbodies, fish habitats). Identification of the sensitive elements on the receiving environment must also be carried out by these studies (e.g., sources of drinking water, major waterbodies, wetlands, archaeology sites). The results of environmental studies will have to be considered to identify project alternatives.

Once the baseline has been defined by the latter studies, an impact assessment study must be carried out to identify and analyze both the positive and negative impacts of the mining project. Mitigation measures are then proposed to attenuate the negative effects on the environment.

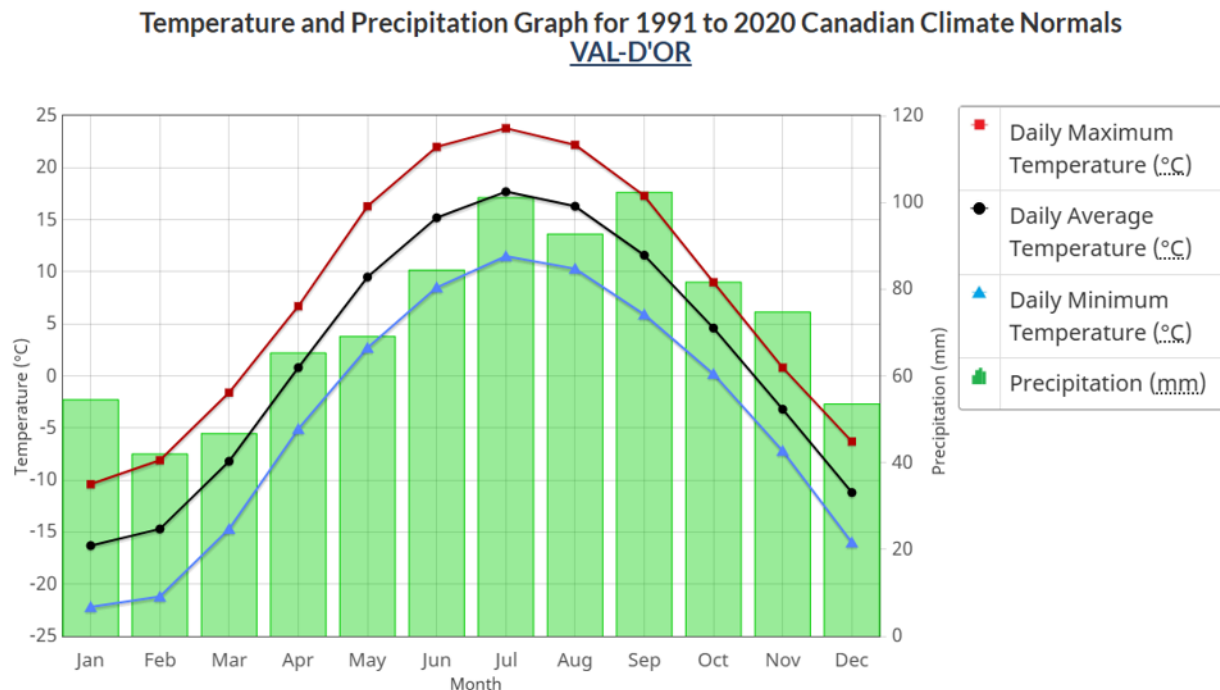
Environmental studies were begun by Stantec in 2026. This desk review study also includes an archaeological assessment and a Limited Phase I Environmental Site Assessment (ESA) report, both prepared by Stantec in 2026. These prior studies covered the area of the Cadillac project delimited by the mining titles (Figure 20-1), referred to as the study area. The next step for the Cadillac project is the execution of the baseline studies to be able to conduct an impact assessment study afterward. Other documents are available from the previous period of exploitation (1989-1997), per example the closure plan (Cambior, 1996) and the dismantling report (Cambior, 1998). Description of the receiving environment and reclamation works are presented in these documents, but several components need to be updated to meet current standards and regulations.



20.1.1.1 Climate



Figure 20-2 : Temperature and precipitation graph at the Val-d'Or weather station (ECCC, 2026)



20.1.1.2 Air Quality

A station from the Réseau de surveillance de la qualité de l'air du Québec (RSQAQ) is in Senneterre, approximately 38 km north from the study area. The analysis of annual hourly proportions indicates an overall excellent air quality at the regional scale (Stantec, 2026). These results indicate a weak influence from local emission sources and is typical of a low-industrial forest environment. Variations were observed in 2021 and 2023 with a temporary increase in fine particulate matter concentrations (PM2.5) caused by wildfires.

20.1.1.3 Hydrology and Surface Water Quality

Hydrology was reviewed by Stantec (2026) relying on several complementary sources of information. The study area is part of the Nottaway River watershed located within the hydrographic region 08 (Hannah and Rupert Bays). At the regional scale, the study area is located within the Bell River sub-watershed (22,222 km²), and within the Louvicourt River sub-watershed (1,464.5 km²) at the local scale. A precise watershed delineation for the Cadillac project is required to accurately model local hydrology and adapt infrastructure design.

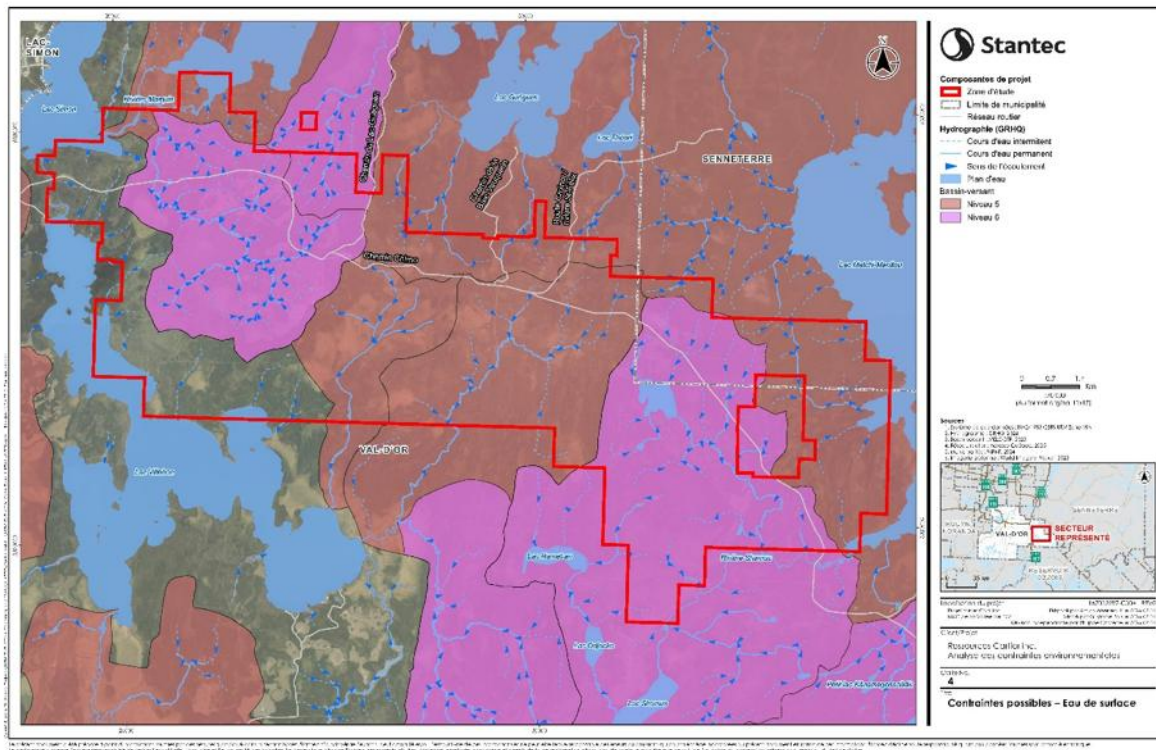
Figure 20-3 presents the main hydrological features. The project area boundaries are defined to the north by Lake Guéguen and Lac Simon, to the east by Lake Matchi-Manitou, and to the south by Lake



Villebon. Waterbodies, including small lakes and marshes, cover a total surface area of 3,756,811 m². The main watercourses are the Marquis River, Villebon River and Shamus River. The study area includes 122 permanent watercourses, 153 intermittent watercourses, and 130 unclassified watercourses, for a total length of 172,786 m (Stantec, 2026). Field characterization is required to verify their actual ground-level presence.

Surface water quality surveys are also required since no previous sampling campaigns have been conducted in the study area. Only two (2) samples of surface water and sediments were collected prior to the construction of the tailings storage facility in 1993 (Cambior, 1996). Chromium and iron levels are the sole parameters to exceed drinking water criteria (CCREM, 1987). Arsenic levels were high in the sediments samples, but the entire mine sector is known to be naturally rich in arsenic (Cambior, 1996).

Figure 20-3 : Hydrological features (Stantec, 2026)



20.1.1.4 Hydrogeology and Groundwater Quality

Review of hydrogeological data was conducted by Stantec (2026). The hydrogeology of the Bell River watershed is dominated by glaciolacustrine clay deposits from Lake Ojibway-Barlow. The main

recharge zones are located within fluvioglacial deposits. The nearest groundwater wells are located in the Lac Simon community and at the Lake Villebon outfitter. An esker has been identified within the study zone; however, the thickness of the deposits and the water quality are unknown. More information is available in the closure plan for the former Chimo mine (Cambior, 1996). Three (3) stratigraphic units were identified: peat, silt and sand. The sand that composes the esker ranges in composition from gravelly sand to fine sand with traces of silt. However, the delineation between the esker and the peat is poorly defined. The possibility of an aquifer within the esker, as well as its potential, need to be investigated in further hydrogeological studies. Characterization of the groundwater quality within the study area is also required.

20.1.1.5 Soils Quality

A Phase I ESA was conducted by Stantec (2026) and is presented in the appendices of the main environmental review report. This study is an initial step in the site environmental assessment process that can lead to a more detailed environmental site assessment (Phase II ESA). Due to the past mining, the forestry, or recreational activities in the area, potential sources of contamination (petroleum products, oils, metals) require further investigation. A soil characterization study following the Guide de caractérisation physicochimique de l'état initial des sols avant l'implantation d'un projet industriel (MDDELCC, 2016) by the Ministère de l'Environnement, de la Lutte contre les changements climatiques, de la Faune et des Parcs (MELCCFP - Ministry of the Environment, the Fight Against Climate Change, Wildlife and Parks) will also be required to assess natural background levels in the study area.

20.1.2 Biological Environment

20.1.2.1 Wetlands

Wetland mapping is available through the *Cartographie des milieux humides potentiels du Québec* (CMHPQ) by the MELCCFP (2023). According to this cartography reviewed by Stantec (2026), the study area is covered at 46% by wetlands : forested swamps (17%), wooded peatlands (16%), open ombrotrophic peatlands (10%), open minerotrophic peatlands (2%), and marshes (<1%). Adverse effects on wetlands of the Cadillac project are unavoidable, so compensation measures will be required pursuant to the Environmental Quality Act (EQA) and the Regulation respecting compensation for adverse effects on wetlands and bodies of water (Q-2, r. 9.1). The actual type and proportions of wetlands will be described during the baseline studies.



20.1.2.2 Terrestrial Habitats

Terrestrial habitats were reviewed by Stantec (2026) using data from the ecoforest map of the *Ministère des Ressources naturelles et des Forêts* (MRNF – Ministry of Natural Resources and Forests, 2026). To the southeast of the study area, the ecological region is the Cabonga Reservoir (4b), located within the balsam fir-yellow birch bioclimatic domain (Stantec, 2026). Region 4b is dominated by mixedwood stands, with presence of hardwood stands (sugar maple, red maple and yellow birch) and softwood stands (balsam fir-black spruce and black spruce-balsam fir). To the northwest, the ecological region is the Abitibi Plain (5a), located within the balsam fir-paper birch bioclimatic domain (Stantec, 2026). Region 5a is characterized by a dominance of softwood stands (black spruce, jack pine, balsam fir), with presence of hardwood stands (trembling aspen, paper birch, red maple).

Stands exclusively composed of hardwoods are rare (3%), while softwood stands (64%) and mixedwood stands (32%) are more abundant. Stands older than 70 years represent less than 20%, due to severe outbreak, wildfire, clearcut, or cutting with regeneration protection.

20.1.2.3 Floristic Species with Status

Species with status, or at risk, are protected by the provincial and federal legislations. The initial review study carried out by Stantec (2026) identified eleven (11) floristic species with status potentially present within the habitats of the study area. All the species are likely to be designated as threatened or vulnerable (provincial status), except for the *Erythranthe geyeri* which is a threatened species. None of them have a status at the federal level. Field inventories in suitable habitats will be required to confirm the presence or absence of these species.

20.1.2.4 Mammals

Several data sources were consulted by Stantec (2026) to provide an overview of land wildlife (big and small mammals), including species with status. Big mammals that are likely to frequent the study area are the black bear, moose, Canada lynx and coyote. According to the species-at-risk occurrence map (CDPNQ, 2023), the study area is partially covered by an occurrence associated with the woodland caribou. This species is designed as vulnerable in Quebec and as threatened in Canada. Nonetheless, the woodland caribou population present in the Val-d'Or region exhibits a low population size and is kept in an enclosure managed by the MELCCFP since 2020 for its protection. For the small mammals, the species identified by the review study are the fisher, North American



porcupine, striped skunk, American marten, woodchuck, snowshoe hare, least weasel, long-tailed weasel, North American beaver, North American river otter and red fox. None of the identified species are species-at-risk (Stantec, 2026). Micromammals, such as shrews, voles and mice, are also likely to be present within the study area.

The study area is included within the range of four (4) chiropteran species : little brown myotis, eastern red bat, hoary bat and the northern myotis. Field inventories are required in the suitable habitats, since all these species have a status at the provincial level, and two (2) of them at the federal level.

20.1.2.5 Herpetofauna

Herpetofauna includes amphibians and reptiles. Range map and databases consulted by Stantec (2026) indicated that eleven (11) species of amphibians and four (4) species of reptiles are likely to frequent the habitats of the study area. None of them are special-status species (Stantec, 2026).

20.1.2.6 Avifauna

The review study done by Stantec (2026) indicates the potential presence of 166 avian species in the study area. This list includes one threatened species, two vulnerable species, and eight species likely to be so designated at the provincial level. Four species hold a federal protective status, but don't have one at the provincial level. A bald eagle breeding site, a species designated as vulnerable in Quebec, is historically reported along the shores of Lake Villebon. Field inventories of avian species will be required as part of the baseline studies.

20.1.2.7 Ichthyofauna

Several data sources were reviewed to characterize the aquatic fauna within the study area (Stantec, 2026). Twenty-two (22) fish species were recorded in the study area's lakes by the MELCCFP. The only special-status species is the lake sturgeon, a species of cultural importance to the First Nation communities who use it for subsistence purposes. Recreational fishing is important in the region, with many outfitters present. Inventories of fish species and their habitats will be required in lakes and watercourses as part of the baseline studies.

20.1.3 Human Environment

The Cadillac project area is in the Abitibi-Témiscamingue administrative region, and the nearest town is Val-d'Or. However, the project area overlaps the territories of the town of Val-d'Or and the



town of Senneterre. The project area lies within the traditional territory of the Lac Simon Anishnabe Nation (*Simosagigan*).

20.1.3.1 Protected Areas

Review of the potential mining restriction zones was completed by Stantec (2026), and additional information was collected by consulting several governmental interactive maps.

There is no protected area, such as biodiversity reserve or exceptional forest ecosystem, in the Cadillac project area (Stantec, 2026). However, there is a proposed biological refuge that partially intersects with the project boundaries (yellow zone, Figure 20-1). This area of 505 ha (08451R053) is at the stage of a proposed area and there is no constraint associated (Stantec, 2026).

At the municipal level, Lake Villebon and Lake Guéguen are located within the protection zones 519-P and 515-P. Extractive industry is not allowed in the protection zone (Stantec, 2026). Since mining activities are not planned within these hydric areas, this constraint is not an issue for the development of the Cadillac project.

The *Commission de protection du territoire Agricole du Québec* (CPTAQ) is responsible for defining the boundaries of agricultural zones. The Cadillac project is located outside the agricultural zone.

20.1.3.2 Cultural Heritage

Material and intangible cultural heritage was reviewed by Stantec (2026). No immovable, site, or landscape is listed within the boundaries of the study area. However, the area is part of the traditional territory of the Anishnabe, whose traditional practices constitute a living heritage that contributes to the understanding of regional human occupation, even though they are not formally listed as intangible cultural heritage (Stantec, 2026).

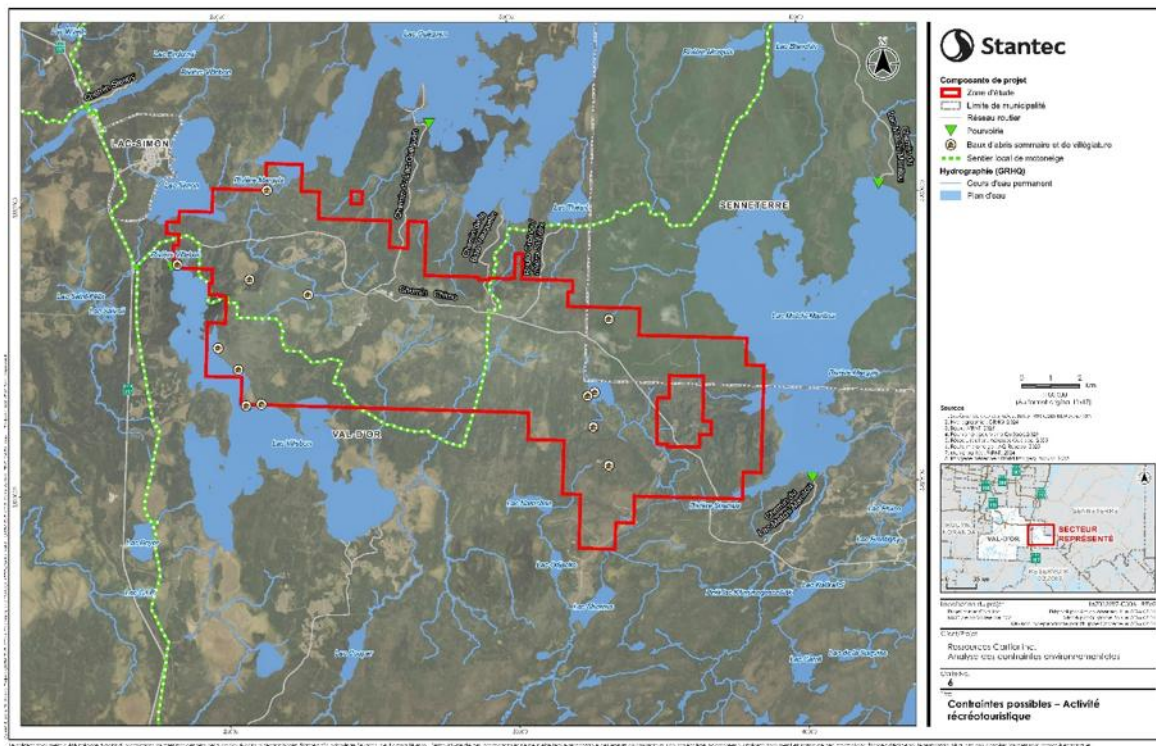
20.1.3.3 Territory and Land Uses

Different land users are present within the Cadillac project site and its surrounding area (Figure 20-4). There are a few past-producing mines and mineral exploration activities, as well as forestry activities. These activities led to the construction of several roads in the region. The former Chimo gold mine is centrally located within the Cadillac project area, and its reopening is part of the new mining project. According to the land-use plan (MRCVO, 2023), the project area is located within the forest land-use zone. This land-use designation is primarily intended for the exploitation of natural forest resources and for recreational and wildlife-related activities. It also permits the extraction of mineral resources, which is authorized without restriction (Stantec, 2026).



Other land uses include traditional and recreational outdoor activities such as hunting, fishing and trapping. Three (3) outfitters are present nearby, leading to an increase in seasonal land use and the frequentation of waterbodies. In the study area, recreational land use is reflected in particular by the presence of six (6) resort leases issued by the MRNF and seven (7) rustic shelters (Stantec, 2026). The study area is crossed by a local snowmobile trail connected to regional trail no.386 and to provincial Trans-Québec trail no.83. These infrastructures are managed by the *Club Motoneige Val-d'Or* (897). There are no trails belonging to the *Fédération Québécoise des Clubs Quads* in the study area. Nevertheless, the presence of numerous roads or unregistered trails in the area suggests potential use by ATV riders.

Figure 20-4 : Territory and land uses in the study area (Stantec, 2026)



20.1.3.4 Transport Infrastructure

The study zone is crossed from west to east by the Chimo road. This is an unpaved, primary multi-use forest road. It provides access to several secondary roads and trails. The Croinor/Rivière-Saint-Félix road, which is also classified as a primary multi-use forest road, provides access to the northern portion of the study zone from Senneterre.

The nearest railway infrastructures are located in Senneterre and Val-d'Or. There is no active railway

infrastructure within the study area. The nearest airport infrastructure is located to the regional airport of Val-d'Or, approximately 35 km northeast of the Cadillac project.

20.1.3.5 Waste Disposal Sites

A satellite ecocentre is located at kilometer 10 of the Chimo road. This facility consists of containers used for the disposal and sorting of residual materials generated by resort activities, including household waste, recyclables, and certain materials such as wood and metal (Stantec, 2026).

20.1.3.6 Archaeological Potential

The archaeological potential in the study area was investigated by Stantec (2026) and is presented in the appendices of the main environmental review report. Significant sites along the shores of Lake Simon demonstrate traditional land use by paleohistoric groups. Other sites discovered at Lakes Matchi-Manitou and Guéguen support the premise that human occupation in the territory was not confined solely around Lake Simon. The hydrographic network has made the area attractive for human occupation during the paleohistoric period. However, no archaeological data is available for the territory between these lakes, which may require more investigation.

20.2 Waste Rock, Tailings and Water Management

At this phase of the Project, several assumptions were made about the geotechnical and geochemical aspects of the waste rock, mineralized material, tailings and water management. Complete studies will be conducted during a subsequent phase. The infrastructures should be designed in accordance with Directive 019 (MELCCFP, 2025).

The layout of the Cadillac project is shown in Figure 18-1. Production is exclusively in underground mines, involving five (5) portals. Each portal site has its own waste rock pile, overburden pile, mineralized material stockpile and water pond. Most of the waste rock will be used as rock fill underground with a limited proportion needed to be stored above ground in waste rock piles. Storage capacities are showed in Table 20-1.



Table 20-1: Storage capacities of waste rock and overburden management facilities

Mine Site (West to East)	Type of Material	Superficial Area (m ²)	Storage Capacity (tonnes)
1	Waste rock	54,625	1,200,000
	Overburden	8,300	60,400
2	Waste rock	126,580	3,070,000
	Overburden	29,000	180,000
3	Waste rock	44,800	650,000
	Overburden	8,550	56,500
4	Waste rock	18,650	285,000
	Overburden	4,825	30,200
5	Waste rock	34,470	565,000
	Overburden	7,335	55,500
TOTAL	Waste rock	279,125	5,770,000
	Overburden	58,010	382,600

Process plant is located near the Chimo mine. There is only one (1) tailings management facility (TMF) built in two (2) phases. Superficial area of the total TMF is 1,080,000 m² with a storage capacity of 9,5 Mt. It was designed for the storage of thickened tailings (70-80 % solids). The TMF was intentionally spread out to reduce geotechnical complexity and the impacts on the landscape. The natural topography was used to optimize drainage. Site selection for the TMF was based on the following criteria:

- Reduce the distance between the TMF and the plant.
- Avoid areas highlighted as a potential mineralized zone.
- Reuse of the former tailings storage facility.
- Use the natural slope for drainage.
- Avoid major watercourses and highly permeable soil.
- Avoid protected or high land-use potential areas.

All mining waters will be flowing in a closed system. That means that every water in contact with the mine site will be collected, sent into basins or recirculated for the plant operations. The final effluent will meet water quality criteria, and the water will be treated, if needed, before being released in the receiving environment.

A full geochemical characterization study for tailings, mineralized material, waste rocks, and possibly overburden classification needs to be completed to evaluate the metal leaching and acid rock drainage (ML-ARD) potential. This study must be done according to the “*MELCCFP Guide de caractérisation des résidus miniers et du minéral*” applicable in Quebec (MELCC, 2020). This



geochemical characterization has started in August 2026 with the sampling of waste rocks and mineralized material by Stantec. Test results are not yet available, and the ML-ARD classification is pending.

20.3 Permitting Requirements

The construction, production and closure of a mine is regulated by several acts and regulations at three different levels: federal, provincial and municipal. The overview of the regulatory context described in this section is based on acts and regulations in place at the time of the preparation of this PEA.

20.3.1 Federal Regulations

20.3.1.1 Impact Assessment Act (S.C. 2019, c.28, s.1) and Physical Activities Regulations (SOR/2019-285)

Under the Impact Assessment Act (IAA), only projects designated by the Physical Activities Regulations are subjected to the environmental assessment procedure. Thus, an environmental assessment under the IAA is required for a project that involves the construction, operation, decommissioning, and abandonment of a new metal mine, other than a rare earth element mine, placer mine or uranium mine, with a production capacity of 5,000 t/d or more.

This is not the case for the Cadillac project. Therefore, it won't be subject to the environmental assessment procedure at the federal level. Nevertheless, the project needs to comply with federal acts and regulations.

20.3.1.2 Fisheries Act (R.S.C., 1985, c. F-14), Metal and Diamond Mining Effluent Regulations (SOR/2002-222) and Authorizations Concerning Fish and Fish Habitat Protection Regulations (SOR/2019-286)

Under the Fisheries Act, the Metal and Diamond Mining Effluent Regulations (MDMER) provide the framework for mining activities regarding the protection of fish and fish habitats. The MDMER sets criteria for mining effluents.

The Authorizations Concerning Fish and Fish Habitat Protection Regulations set out the process under the Fisheries Act for obtaining authorization to carry out works, undertakings, or activities that may cause the death of fish or the harmful alteration, disruption, or destruction of fish habitat.



20.3.1.3 Other Federal Acts or Regulations

A mining project must comply with other federal acts or regulations. For example, they are subject to the Species at Risk Act (S.C. 2002, c. 29), the Migratory Birds Convention Act (S.C. 1994, c. 22), the Explosives Act (R.S.C., 1985, c. E-17), the Hazardous Products Act (R.S.C., 1985, c. H-3), the Transportation of Dangerous Goods Act (1992), the Canadian Navigable Waters Act (R.S.C., 1985, c.N-22), the Canadian Environmental Protection Act (S.C. 1999, c.33), and all regulations under these acts that may apply.

20.3.2 Provincial Regulations

20.3.2.1 Environmental Quality Act (CQLR, c. Q-2)

The Environmental Quality Act (EQA) is the main environmental protection law in Quebec. It leads to numerous regulations that aim to protect the environment, including its ecological, social, and economic dimensions. Under the EQA, the Regulation Respecting the Environmental Impact Assessment and Review of Certain Projects (Q-2, r.23.1) establishes the requirements and procedure for impact assessment of development projects in meridional Quebec. Projects listed under Schedule 1 of the Regulation are subject to the provincial impact assessment and review procedure. However, all activities required for the operation of a new mine are subject to the procedure since November 2024.

The procedure for the emission of a ministerial decree involves several phases, including public consultation, analyses of the impact study and an eventual review by the *Bureau d'audiences publiques sur l'environnement* (BAPE). After that, authorizations under Article 22 of the EQA must be obtained for the Project activities.

The MRNF published a document that provides an exhaustive list of permits required (MRNF, 2023). The Cadillac project will have to conform with the requirements of the latest version of Directive 019 on the Mining Industry (MELCCFP, 2025).

20.3.2.2 Mining Act (CQLR, c. M-13.1)

The Mining Act provides the regulatory framework for mining exploration permits and titles (including the mining lease), the rehabilitation and restoration plan, and the financial guarantee. To obtain the mining lease, the Project's survey plan must be formally approved by the Office of the Surveyor-General of Quebec, the rehabilitation and restoration plan must be approved by the MRNF and the authorization under the EQA must be issued. More details about the closure plan are described in



section 20.5. Within the 30 days after the mining lease is issued, the company must establish a monitoring committee to foster the involvement of the local community in the project.

In November 2024, the Bill 63, an Act to amend the Mining Act and other provisions was assented. It made all new mine projects subject to the environmental impact assessment and review procedure by the EQA.

20.3.3 Municipal Regulations

The municipal regulations include local municipalities and regional county municipality (RCM) authorities. The Cadillac project area is part of the town of Val-d'Or and the town of Senneterre, which are both part of the RCM of La Vallée-de-l'Or. The Cadillac project will need to comply with municipal regulations of these authorities.

20.4 Social or Community Requirements

One of the early steps of social requirements involves identifying the various stakeholders who are likely to be affected or concerned with the mining project. Stakeholders are individuals, groups, or organisations who could be directly or indirectly affected by the project or has an interest in its outcome. Consultations with stakeholders should be initiated early in the project development to ensure that the concerns and issues that could be raised are clearly identified to take them into consideration in the project. Consultation meetings with stakeholders and First Nation authorities are an opportunity for participants to ask questions, share their comments and express their concerns about the project.

An engagement process will be implemented by Cartier to meet various stakeholders concerned by the Cadillac project, First Nations communities, land users, development economic organizations, environmental groups and academic community. Talks between Cartier and the Anishnabe Nation Council of Lac-Simon have been initiated since the beginnings of exploration works by Cartier in the region.

20.5 Mine Closure Requirements

A land rehabilitation and restoration plan ("closure plan") and a financial guarantee are required under the provincial Mining Act (m-13.1, article 232). Guidelines for preparing mine closure plan in Quebec is provided by the MRNF. Rehabilitation involves all activities after mining operations to



return the site affected by mining activity to a satisfactory condition. The latest version of this guide (MRNF, 2024) defines the aim to a satisfactory condition by:

- Eliminating unacceptable health hazards and ensuring public safety.
- Limiting the production and circulation of substances that could damage the receiving environment, and trying to eliminate any form of maintenance and monitoring in a long term.
- Restoring the site to a condition that approaches the initial or surrounding ecosystem, including tailings and mine waste rock accumulation areas where possible.
- Restoring the infrastructure areas to a condition compatible with future use.

The closure plan must be approved by the Minister before the issue of the mining lease. A financial guarantee to fully implement the closure plan must be provided in three (3) payments in the first two (2) years following its approval. The closure plan must be revised every five (5) years, unless a shorter period is fixed by the Minister.

Finally, monitoring is required during the period specified by the approved plan to ensure that the rehabilitation has been successfully completed. When the Minister declares to be satisfied with the rehabilitation and restoration work carried out, a certificate attesting that the operator has been released from its obligations is issued.

General design of the TMF and the waste rock piles allows progressive restoration work on the Cadillac project. Overburden stocked next to the waste rock piles or the TMF could be used for the restoration work. At the end of mine life, all buildings will be dismantled, the soil will be leveled, topsoil will be replaced on it, and it will be revegetated. Estimated closure costs have been benchmarked against recent mining projects in similar jurisdictions. Costs could be estimated at approximately C\$40 million.

Closure plan of the former Chimo mine was prepared by Cambior in 1996. Closure work has been carried out in accordance with the requirements of the MRNF, and the site was considered free of risks for the environment, human health and safety. To conduct exploration works on the reclaimed tailings pond, Cartier must submit a closure plan to the MRNF under the same conditions as those for mining operations. Other exploration works are subject to a closure plan submittal:

- Surface mineral sampling exceeding 500 t.
- Displacing more than 5,000 m³ of soils, or stripping a continuous surface area of 10,000 m² or more.



- Underground activities, such as exploration ramps of galleries, sinking shafts and mine dewatering.
- Setting up facilities for the storage of mineralized material or waste rock.

The provisions of the Mining Act do not affect or restrict the application of the EQA. It is the applicant's responsibility to verify with the MELCCFP and any other municipal or government entity whether other authorizations are required to carry out these activities (MRNF, 2024).



21 CAPITAL AND OPERATING COSTS

21.1 Basis of Estimate

The base date of the capital and operating cost estimates is the third quarter (“Q3”) of 2026. The pre-production period is expected to last 24 months. The cost estimates are aligned with an owner-managed project delivery model with toll milling at the start of production.

The majority of costs will be incurred in Canadian dollars. The costs estimated are presented in Canadian dollars and no allowance for escalation or exchange rate fluctuation were used. An exchange rate of 1.38 CAD per 1.00 USD has been applied when converting United States dollars into Canadian dollars.

21.2 Total Capital Costs

Life-of-mine project capital costs are estimated to total \$1,685.1 million gross of pre-production revenue (\$1,654.9 million net of pre-production revenue), split in the following the following categories:

- Initial Capital Costs – This category includes all costs to develop the property and does not include a process plant, as the project starts with a toll mining operation for the first year of production. Initial capital costs total \$306.3 million (including \$37.9 million for contingency but gross of \$30.6 million in pre-production revenue), which will be expended over a 24-month period of construction, pre-production and commissioning.
- Growth capital costs – This category includes all costs related to the construction of process plant expended during the first year of production, as well as the expansion of this plant during the fourth year of production. Growth capital costs are estimated to be \$277.1 million, inclusive of a \$37.8 million contingency.
- Sustaining Capital Costs – This category includes all costs related to the acquisition, replacement, or major overhaul of assets required to sustain operations, as well as underground development and infrastructure during the production phase of the Project. Sustaining capital costs are estimated to be \$1,062.0 million, inclusive of a \$138.5 million contingency.



- Working Capital – This category consists of costs related to building up an inventory. It is planned to maintain a supplies inventory equivalent to 45 days of operating expenses, as well as a finished goods inventory equivalent to 7 days of revenue. Additionally, it is planned to pay the various operational suppliers within 30 days, and 7 days of accounts receivable are considered. Though the total net changes in working capital over the life of mine is null, the maximum amount of working capital for the Project is \$34.9 million.
- Closure Costs – This category includes all costs related to the closure, reclamation, and ongoing monitoring of the mine after operations. Closure costs are estimated at \$40.0 million.

The total capital costs are summarized in Table 21-1.

Table 21-1: Capital Costs Summary

Cost Area	Initial Capital Costs (\$M)	Growth Capital Costs (\$M)	Sustaining Capital Costs (\$M)	Total Capital Costs (\$M)
Infrastructure	40.9	-	52.5	93.4
Electrical and communication	27.9	-	8.1	36.0
Waste and water management	9.2	-	58.7	68.0
Mine	91.3	-	720.2	811.5
Process plant	9.0	230.2	-	239.2
Indirect costs	29.8	9.0	84.0	122.7
Contingency	37.9	37.8	138.5	214.2
Capitalized operating costs	60.0	-	-	60.0
Closure	-	-	40.0	40.0
Total	306.0	277.1	1,102.0	1,685.1
Pre-production revenue	-30.2	-	-	-30.2
Total (net of pre-production revenue)	275.8	27.1	1,102.0	1,654.9



21.3 Initial Capital Costs

21.3.1 Initial Capital Cost Summary

The initial capital costs are estimated at \$306.0 million gross of pre-production revenue and \$275.8M net of pre-production revenue. Pre-production revenue will amount to \$30.2 million from 6 koz of gold recovered. The initial capital expenditures include a contingency of \$37.9 million. A summary of the initial capital expenditures is presented in Table 21-2.

Table 21-2: Initial Capital Costs Summary

Cost Area	Initial Capital Costs (\$M)
Infrastructure	40.9
Electrical and communication	27.9
Waste and water management	9.2
Mine	91.3
Process plant	9.0
Indirect costs	29.8
Contingency	37.9
Capitalized operating costs	60.0
Closure	-
Total	306.0
Pre-production revenue	-30.2
Total (net of pre-production revenue)	275.8

21.3.2 Infrastructure Initial Capital Costs

The initial capital expenditures cost for infrastructure totals \$40.9 million and is summarized in Table 21-3. The main elements of this cost area are the mobile equipment maintenance facility, the warehouse, the office and mine dry building and the surface infrastructure supporting underground operations such as the mine ventilation fans and compressors. The detailed description of planned infrastructure is presented in Item 18.

Table 21-3: Infrastructure Initial Capital Costs

Cost Area	Initial Capital Costs (\$M)
Infrastructure	1.0
Roads, bridges and fencing	16.5
Buildings	23.0
Fuel storage	0.4
Total	40.9



21.3.3 Electrical and Communications Initial Capital Costs

The initial capital cost estimate for the electrical and communications cost area totals \$27.9 million and is summarized in Table 21-4. The main elements of this cost area include the construction of a 120 kV electrical powerline to the existing line along Quebec Route 117 and the mine site along the existing 25 kV powerline, as well as a 120 kV / 13.8 kV substation on site.

Table 21-4: Electrical and Communications Initial Capital Costs

Cost Area	Initial Capital Costs (\$M)
Electrical powerlines and distribution	15.6
Electrical substation (120 kV / 13.8 kV)	8.0
Communications	2.3
Emergency power generation	2.0
Total	27.9

21.3.4 Waste and Water Management Initial Capital Costs

The initial capital cost estimate for the waste and water management cost area totals \$9.2 million and is summarized in Table 21-5. The main elements of this cost area are water management infrastructure, including ditches, a contact water pond and a water treatment plant.

Table 21-5: Waste and Water Management Initial Capital Costs

Cost Area	Initial Capital Costs (\$M)
Fresh water intake	0.3
Water ponds, ditches and water management	6.8
Waste management facilities	1.9
Overburden management facilities	0.3
Total	9.2

21.3.5 Mine Initial Capital Costs

The initial capital cost estimate for the mine cost area is summarized in Table 21-6. The costs éléments in this category include underground mine development, infrastructure and equipment. The total cost was estimated at \$91.3 million.



Table 21-6: Mine Initial Capital Costs

Cost Area	Initial Capital Costs (\$M)
UG Mobile equipment	22.1
Surface Mobile equipment	1.7
Surface Infrastructure	2.9
Surface Fixed equipment	2.8
UG Fixed equipment	0.7
UG Infrastructure	6.5
UG Lateral development	51.8
UG Vertical development	1.9
UG Small equipment	0.9
Total	91.3

21.3.6 Process Plant Initial Capital Costs

For the process plant cost area, only earthworks and preparatory work are planned for the pre-production period of the project. These amount to \$9.0 million. With the first year of operations considering toll mill processing, the main construction of the on-site process plant is planned for Year 1.

21.3.7 Indirect Costs, Contingency, Capitalized Operating Costs and Pre-Production Revenue

As part of overall initial capital costs, construction indirect cost were calculated as a 34% allowance over all direct initial capital costs other than those of mining, totaling \$29.8 million. A 32% allowance was considered for contingency, calculated on the sum of all direct initial capital cost other than mining, plus indirect costs. All operating costs incurred during the pre-production period were capitalized, totaling \$60.0 million. Revenue realized during the pre-production period was capitalized as well, net of selling costs and royalties, for a total of \$30.2 million.

21.4 Growth Capital Costs

21.4.1 Process Plant Growth Capital Costs

The growth capital cost estimate for the process plant cost area is summarized in Table 21-7, and are associated with the construction of a 3,000 tonne per day processing facility during year 1, and the expansion of the facility to approximately 4,250 tonnes per day during year 4.



Table 21-7: Process Plant Growth Capital Costs

Cost Area	Construction (\$M)	Expansion (\$M)	Growth Capital Costs (\$M)
Machinery and Equipment	29.2	7.7	36.9
Piping	1.3	0.3	1.6
Valves	0.6	0.2	0.7
Electricals	2.7	0.7	3.4
Instrumentation	0.4	0.1	0.5
General Earthworks	4.8	1.3	6.1
Civil works (structural concrete, supply & Installation)	35.8	9.5	45.3
SMP Steel structures (Supply and Installation)	24.7	6.6	31.3
SMP Tanks (Supply and Installation)	9.7	2.6	12.3
SMP Plate works (Supply and Installation)	7.5	2.0	9.5
SMP Mechanical Assembly	15.5	4.1	19.6
SMP Piping installation	21.7	5.7	27.4
E&I Installation	22.3	5.9	28.2
Plant Associated Facilities (offices, stores, etc.)	12.9	3.4	16.3
Total	189.1	50.1	239.2

21.4.2 Indirect and Contingency Growth Capital Costs

As part of overall growth capital costs, construction indirect costs were estimated at \$9.0 million, being relatively limited due to the mine being in operation during the main part of process plant construction and owner's cost being incurred as operating costs. A 16% allowance was considered for contingency, calculated on the sum of direct initial capital cost and indirect costs.

21.5 Sustaining Capital Costs

Sustaining capital costs are estimated at \$1,062.0 million, inclusive of \$84.0 million in indirect costs and a contingency of \$138.5 million. These are mostly associated with the construction of surface infrastructure for portals areas developed during the production phase of the project, as well as mining development, mining infrastructure and mining mobile equipment in support of maintaining production capacity over the life of mine. A summary of the sustaining capital costs is presented in Table 21-8.

Table 21-8: Sustaining Capital Costs Summary



Cost Area	Sustaining Capital Costs (\$M)
Infrastructure	52.5
Electrical and communications	8.1
Waste and water management	58.7
Mine	720.2
Indirect costs	84.0
Contingency	138.5
Total	1,062.0

21.5.1 Closure Costs

The closure costs are estimated to be \$40.0 million. Closure costs cover the following activities:

- Scarification, regrading, growth material placement and seeding of the areas in which roads were present
- Regrading, growth material placement and seeding of the waste rock management facilities
- Regrading, growth material placement and seeding of the overburden management facilities
- Capping of underground ventilation raises and portals
- Dismantlement and salvaging of buildings and transmission line
- Regrading, growth material placement and seeding of sites in which buildings were present
- Post-closure monitoring and reporting

21.6 Working Capital

Working capital consists of costs related to building up an inventory. It is planned to maintain a supplies inventory equivalent to 45 days of operating expenses, as well as a finished goods inventory equivalent to 7 days of revenue. Additionally, it is planned to pay the various operational suppliers within 30 days, and 7 days of accounts receivable are considered. Though the total net changes in working capital over the life of mine is null, the maximum amount of working capital for the Project is \$34.9 million.

21.7 Operating Costs

Operating costs are summarized in Table 21-9 and presented by year in Table 21-10. The operating costs include mining, processing, waste and water management and general services and



administrative (“G&A”) costs. The average life of mine operating cost is US\$1,481/oz of gold payable, including selling and royalty costs. The average life of mine all-in sustaining cost (“AISC”) is US\$2,107/oz of gold payable.

Table 21-9 : Operating Costs Summary

Cost Area	Total Cost (\$ M)	Unit Cost		Unit Cost (US\$/oz payable)
Underground mining	2,193.7	95.1	\$/tonne processed	996.0
Off-site transport	15.1	0.7	\$/tonne processed	6.8
Off-site processing	61.8	2.7	\$/tonne processed	28.1
On-site processing	610.5	26.5	\$/tonne processed	277.2
Waste and water management	34.6	1.5	\$/tonne processed	15.7
General and administrative	232.5	10.1	\$/tonne processed	105.6
Subtotal	3,148.2	136.4	\$/tonne processed	1,429.4
Selling costs	11.1	0.5	\$/tonne processed	5.0
Royalty (1.5%)	101.7	4.4	\$/tonne processed	46.2
Total	3,260.9	141.3	\$/tonne processed	1,480.6
Growth capital	277.1	12.0	\$/tonne processed	125.8
Sustaining capital (incl. closure costs)	1,102.0	47.8	\$/tonne processed	500.4
All-in Sustaining Costs (“AISC”)	4,640.0	201.1	\$/tonne processed	2,107.0





Table 21-10 : Total Operating Costs Summary by year

Cost Area	Unit	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18
Underground mining	\$ M	2,193.7	90.3	107.9	110.6	118.6	129.0	148.0	148.2	149.2	151.6	154.5	151.1	148.3	147.5	140.9	147.7	132.1	18.1	-
Off-site transport	\$ M	15.1	15.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Off-site processing	\$ M	61.8	61.8	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
On-site processing	\$ M	610.5	-	33.4	33.5	33.4	39.9	42.1	42.2	42.1	42.1	42.1	42.2	42.1	42.1	42.1	42.2	42.1	6.8	-
Waste and water management	\$ M	34.6	1.5	1.6	1.6	1.6	2.2	2.3	2.4	2.3	2.3	2.3	2.4	2.3	2.3	2.3	2.4	2.3	0.2	-
General and administrative	\$ M	232.5	13.4	13.4	13.5	13.9	14.5	14.6	14.6	14.6	14.6	14.6	14.6	14.6	14.6	14.6	14.6	14.6	3.6	-
Subtotal	\$ M	3,148.2	182.1	156.4	159.2	167.5	185.6	207.0	207.3	208.2	210.6	213.5	210.3	207.3	206.5	199.9	206.8	191.1	28.7	-
Selling costs	\$ M	11.1	0.5	0.7	0.7	0.6	0.7	0.7	0.8	0.7	0.7	0.7	0.7	0.8	0.7	0.6	0.6	0.7	0.1	-
Royalty (1.5%)	\$ M	101.7	5.0	6.0	5.9	5.4	6.2	6.2	7.1	6.9	6.6	6.2	4.5	8.9	6.1	5.6	6.5	8.5	-	-
Total	\$ M	3,260.9	187.6	163.0	165.8	173.6	192.5	213.9	215.3	215.8	218.0	220.5	215.4	217.0	213.3	206.2	214.0	200.3	28.8	-
Growth capital	\$ M	277.1	226.9	-	-	50.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining capital (incl. closure costs)	\$ M	1,102.0	116.1	72.2	90.4	107.8	67.4	75.4	57.9	54.6	50.5	63.1	60.8	45.1	53.1	69.3	44.5	33.6	40.3	-
All-in Sustaining Costs (“AISC”)	\$ M	4,640.0	530.7	235.2	256.1	331.5	259.9	289.3	273.2	270.4	268.4	283.6	276.3	262.1	266.4	275.5	258.4	233.9	69.1	-



21.7.1 Mining Operating Costs

A detailed mine cost build up was developed from basic cost elements such as remuneration costs, consumable prices, fuel prices and equipment productivities.

Equipment operating costs were estimated for each equipment model, which includes operation and maintenance labour, parts (maintenance and repairs), fuel consumption, lubricant consumption, ground engaging tools or tires if applicable. Equipment operating costs were determined from various sources including primarily information from the major suppliers and benchmarked costs from operations in similar environments. Table 21-11 presents the mining costs by cost center.

Table 21-11: Mining Operating Costs Summary

Cost Center	Total (\$ M)	Unit cost (\$/tonne processed)
UG Definition drilling	113.4	4.9
UG Primary opening drilling	46.7	2.0
UG Production drilling	65.6	2.8
UG Blasting	187.9	8.1
UG Mucking	274.0	11.8
UG Hauling	253.7	11.0
UG Backfilling	336.3	14.5
UG Development drilling	55.6	2.4
UG Ground support	168.4	7.3
UG Development services	90.4	3.9
UG Cable bolting	18.0	0.0
UG Shotcreting	10.5	0.8
UG Mine services	418.5	0.5
Surface services	49.0	18.1
Management	45.3	2.1
Grade control	7.1	2.0
Technical services	53.3	0.3
Total	2,193.7	95.1

21.7.2 Processing Operating Costs

Table 21-12 presents the processing costs by cost center.



Table 21-12: Processing Operating Costs Summary

Cost Center	Total (\$ M)	Unit cost (\$/tonne processed)
Off-site transportation	15.1	0.7
Off-site processing	61.8	2.7
On-site processing	610.5	26.5
Total	687.4	29.8

The total processing operating costs estimated amounts to \$687.4 million, which translates to \$29.8 per tonne processed.

21.7.3 Waste and Water Management Operating Costs

The water treatment operating costs were estimated based on the proposed water management strategy for the Project and include labour, reagents, power consumption, monitoring, maintenance, sludge handling, and routine operation of the water treatment facilities. The estimates were developed using design criteria, projected water flows and treatment requirements, supplier budget information and benchmarking against comparable mining operations in Québec. The operating cost assumptions reflect the expected treatment requirements during operations under the planned site water management approach. The waste and water management operating costs for the project were estimated to amount to \$34.6 million over the life of mine, or \$1.5 per tonne processed.

21.7.4 General and Administrative Operating Costs

The general and administrative (“G&A”) operating costs were developed from a detailed bottom-up cost model prepared specifically for the Project and based on the anticipated organizational structure, staffing requirements, site services, and administrative support functions required for the proposed operation. Labour costs were established using projected personnel requirements and current industry salary benchmarks for comparable mining operations in Québec, while non-labour costs were estimated from budget quotations, historical operating data and benchmarking against similar projects. The G&A costs were scheduled over the life of mine and adjusted to reflect the anticipated evolution of the Project through the production phases. They were estimated to total \$232.5 million over the life of mine, or \$10.1 per tonne processed. Table 21-13 provides a summary of G&A operating costs per cost center.



Table 21-13: General and Administrative Operating Costs Summary

Cost Center	Total (\$ M)	Unit cost (\$/tonne processed)
Human resources	23.9	1.0
Health and safety	10.8	0.5
Information technology	7.6	0.3
Environment	35.8	1.6
Administration	41.6	1.8
Surface services	72.0	3.1
Accounting	13.8	0.6
Supply chain	14.1	0.6
Security	12.9	0.6
Logistics	0.0	0.0
Total	232.5	10.1

21.7.5 Workforce

The total workforce for the Cadillac Project, which includes personnel for the mining operation, transport to the toll milling facility, and general and administrative activities is estimated at 515, as detailed in Table 21-14.

Table 21-14: Total Workforce Estimate

Position	Maximum quantity
Mining	387
Processing	92
General and Administrative	36
Total	515

21.7.5.1 Mining Workforce Estimate

The workforce requirement for the mine is projected to be up to 387 persons, as detailed in Table 21-15. This includes positions in operations, supervision, maintenance and technical services.



Table 21-15: Mining Workforce Estimate

Department	Maximum quantity
Operations	286
Supervision	18
Maintenance	26
Technical services	57
Total	387

21.7.5.2 Processing Workforce Estimate

The workforce requirement for the processing items for the Project is projected to peak at 92 persons, as detailed in Table 21-16. This workforce includes roles in management, metallurgy, operations, maintenance, electrical, laboratory and tailings storage facility sectors.

Table 21-16: Processing Workforce Estimate

Department	Phase 1	Phase 2
Management	4	4
Metallurgy	6	6
Operation	30	30
Maintenance	20	22
Electrical Department	10	12
Laboratory	10	10
Tailings Storage Facility (TSF)	8	8
Total	88	92

21.7.5.3 General and Administrative Workforce Estimate

The workforce requirement for general and administrative items for the Project is projected to peak at 36 persons, as detailed in Table 21-17. This workforce includes roles in management, human resources, environment and community relations, IT, health and safety, accounting, purchasing and site maintenance.



Table 21-17: General and Administrative Workforce Estimate

Department	Maximum quantity
Administration	2
Human Resources	3
Health and Safety	2
Accounting	4
Information Technology	1
Supply Chain	3
Surface Services	14
Environment	3
Security	4
Total	36



22 ECONOMIC ANALYSIS

22.1 Overview

The economic and financial evaluation presented in this Technical Report utilizes a discounted cash flow method, both on a pre-tax and after-tax basis. The gold price used in the evaluation was US\$3,600 per troy ounce, as discussed in Item 19 of this Technical Report. The financial model provides results in terms of NPV, payback period, and IRR for the Cadillac project. The economic analysis is conducted in real terms, without considering inflation factors, using Q3 2026 Canadian dollars. The analysis does not take into account project financing.

The economic model estimates cash flows on an annual basis for the life of the Cadillac project, based on the level of engineering and design appropriate for a PEA.

Cash flow projections for the life of the mine are based on sales revenue, operating costs, capital expenditures and other cost estimates. Capital costs are estimated in 5 categories: initial capital, growth capital, sustaining capital, closure and working capital. Operating cost estimates include labour, reagents, maintenance, supplies, services, fuel, and power. Other costs, such as royalties, depreciation, and taxes, are estimated based on the current project and processing plans.

The economic results are calculated from the start of the initial capital expenditures.

22.2 Cautionary Statements

The results of the economic analyses discussed in this section represent forward-looking information as defined under the Canadian securities law. These results are subject to known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here. The forward-looking information includes, but is not limited to, the following:

- Assumed price for gold.
- Cost inflation.
- The proposed mine production plan.
- Assumptions regarding mining dilution and mining recovery.
- The recovery rates of gold in the processing plant.
- Proposed sustaining and operating costs.



- Labour and materials availability.
- Labour and materials costs being approximately consistent with the assumptions in the report.
- Assumptions regarding closure costs.
- Assumptions regarding environmental, social, and licensing risks.
- Changes to tax rates.
- Unexpected variations in the amount of mineralized material and material grade.
- Geotechnical or hydrogeological considerations during mining that differ from the assumptions.
- Ability to maintain social license to operate.
- Unrecognized environmental risks.
- Unforeseen reclamation expenses.
- Failure of plant, equipment, and processes to operate as anticipated.
- The absence of significant disruptions affecting the development and operation of the Cadillac Project.

The availability of certain consumables and services, and the prices for electricity and other key supplies being approximately consistent with the assumptions in the Technical Report.

22.3 Key Assumptions

22.3.1 Gold Price

The determination of gold prices is described in Section 19. The gold price assumption used in the PEA is US\$3,600 per troy ounce, in line with analyst consensus commodity price forecasts.

22.3.2 Other Assumptions

The other key assumptions used in economic analysis are as follows:

- Discount rate of 5%.
- All cost estimates are in constant Q3 2026 Canadian dollars with no inflation or escalation factors considered.



22.4 Metal Production and Revenues

Payable gold over the Project life is 1,602 koz based on an average metallurgical recovery of 94.8% and a payability factor of 99.5%. Payable gold during pre-production is 6 koz, generating estimated revenue of \$30.6 million (net of transportation, refining and royalty costs) which offsets pre-production capital expenditures. A total of 1,596 koz of gold will be payable during operations and will generate net revenue of \$7,918 million. Table 22-1 shows the processing schedule summary while Figure 22-1 presents the yearly gold payable over the LOM.



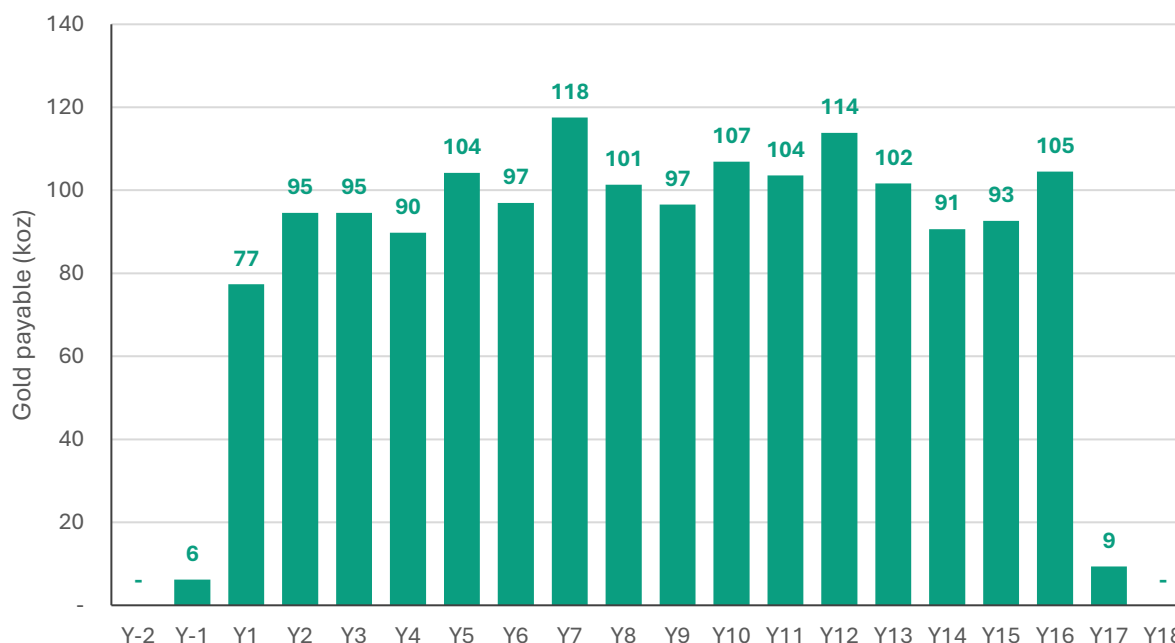


Table 22-1: Processing schedule summary

Item	Unit	Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18
Total mineralized material processed	kt	23,145	-	69	1,005	1,095	1,098	1,095	1,448	1,564	1,569	1,564	1,564	1,564	1,569	1,564	1,564	1,564	1,569	1,564	115	-
Head grade	g/t	2.3	-	3.1	2.7	2.8	2.8	2.7	2.4	2.0	2.5	2.1	2.0	2.2	2.2	2.4	2.1	1.9	1.9	2.2	2.7	-
Gold in head feed	koz	1,698	-	7	86	99	99	95	110	103	124	107	102	113	110	120	108	96	98	111	10	-
Gold recovered	koz	1,610	-	6	78	95	95	90	105	97	118	102	97	107	104	114	102	91	93	105	9	-
Gold payable	koz	1,602	-	6	77	95	95	90	104	97	118	101	97	107	104	114	102	91	93	105	9	-



Figure 22-1 : Gold payable over life of mine



22.5 Royalties

The Cadillac Project is subject to multiple royalty agreements. Parts of the property are subject to either a 1.0% Net Smelter Return royalty, a 2.0% Net Smelter royalty or a 3.0% Gross Metal Royalty. A total of \$101.7 million in royalties was estimated over the life of mine for this Preliminary Economic Assessment.

22.6 Capital Costs

The capital costs considered include initial capital expenditures, growth capital expenditures, sustaining capital expenditures, working capital and closure costs, as detailed below and in Item 21 of this Technical Report.

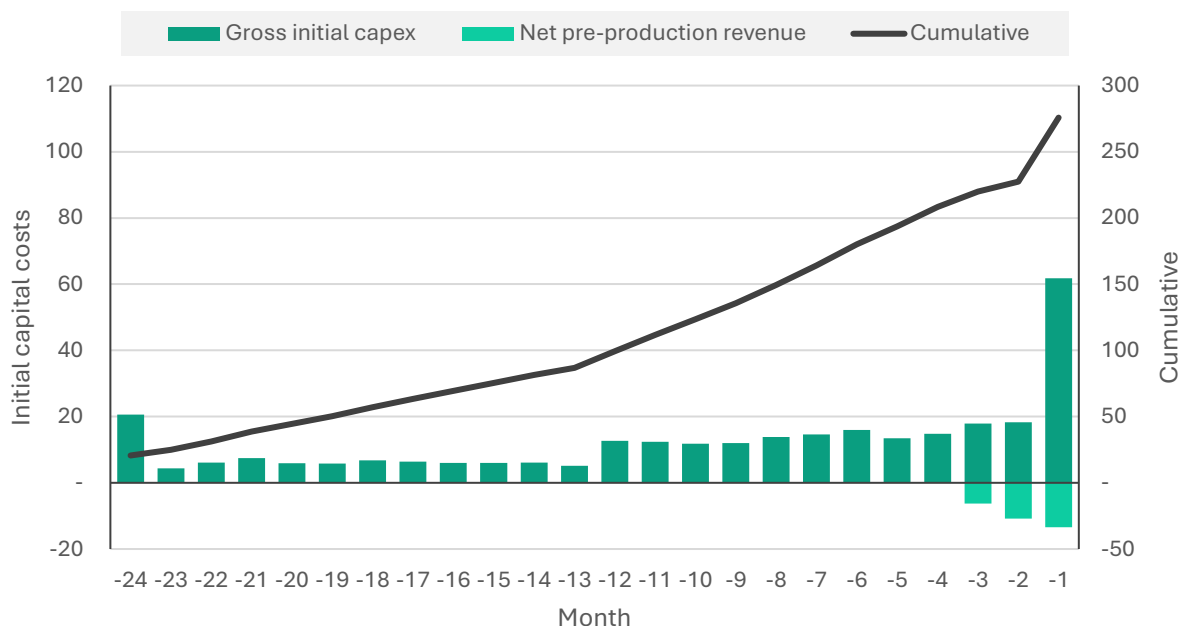
22.6.1 Initial Capital Costs

The initial capital expenditures for Project construction, including pre-production mining, underground development, surface infrastructure construction and other direct and indirect costs is estimated to be \$306.3 million, or \$275.8 million when expressed net of \$30.6 million in pre-production revenue. The total initial capital expenditures include a contingency of \$37.9 million. The



monthly initial capital expenditure estimate is presented in Figure 22-2.

Figure 22-2 : Initial capital expenditures by preproduction month



22.6.2 Growth Capital Costs

Growth capital expenditures are associated with the initial construction of the on-site process plant in year 1 of the project, as well as its expansion of its capacity in year 4 of the project. The total growth capital costs for the Project are estimated at \$277.1 million.

22.6.3 Sustaining Capital Costs

Sustaining capital expenditures are required for establishing surface infrastructures areas in support of the development of new mining zones during the life of mine, as well as the continued development of the underground mine to maintain production. This includes infrastructure construction, equipment purchases, capital spares, major equipment overhauls, development work, indirect costs and associated contingency. The total sustaining capital for the Project is estimated at \$1,062.0 million.

22.6.4 Working Capital

For calculating working capital requirements, a finished goods inventory turnover period of 7 days was considered, as well as a supplies inventory equivalent to 45 days of operating expenses. Additionally, it is planned to pay the various operational suppliers within 30 days, and 7 days of accounts receivable



are considered. With these considerations, the maximum amount of working capital for the Project is \$34.9 million.

22.6.5 Closure Costs

Reclamation and closure costs include infrastructure decommissioning, site shaping and revegetation, maintenance and post closure monitoring. The total reclamation and closure cost is estimated at \$40.0 million, as discussed in Section 21. No residual value was considered.

22.7 Operating Costs

Total operating expenses are presented by year in Table 22-2, and unit operating costs by year in Table 22-3. The operating costs include underground mining, processing and general & administrative costs. The average LOM operating cost (inclusive of selling costs and royalties) is US\$1,481/oz of payable gold or CA\$141/t processed. The average LOM all-in sustaining cost (AISC) is US\$2,107/oz of payable gold.





Table 22-2: Operating costs summary

Cost area	Unit	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18
Underground mining	\$M	2,193.7	90.3	107.9	110.6	118.6	129.0	148.0	148.2	149.2	151.6	154.5	151.1	148.3	147.5	140.9	147.7	132.1	18.1	-
Processing - Mineralized material transportation	\$M	15.1	15.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Processing – Toll milling	\$M	61.8	61.8	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Processing – On-site milling	\$M	610.5	-	33.4	33.5	33.4	39.9	42.1	42.2	42.1	42.1	42.1	42.2	42.1	42.1	42.1	42.2	42.1	6.8	-
Waste and water management	\$M	34.6	1.5	1.6	1.6	1.6	2.2	2.3	2.4	2.3	2.3	2.3	2.4	2.3	2.3	2.3	2.4	2.3	0.2	-
General and administrative	\$M	232.5	13.4	13.4	13.5	13.9	14.5	14.6	14.6	14.6	14.6	14.6	14.6	14.6	14.6	14.6	14.6	14.6	3.6	-
Subtotal	\$M	3,148.2	182.1	156.4	159.2	167.5	185.6	207.0	207.3	208.2	210.6	213.5	210.3	207.3	206.5	199.9	206.8	191.1	28.7	-
Selling costs	\$M	11.1	0.5	0.7	0.7	0.6	0.7	0.7	0.8	0.7	0.7	0.7	0.7	0.8	0.7	0.6	0.6	0.7	0.1	-
Royalties	\$M	101.7	5.0	6.0	5.9	5.4	6.2	6.2	7.1	6.9	6.6	6.2	4.5	8.9	6.1	5.6	6.5	8.5	-	-
Total	\$M	3,260.9	187.6	163.0	165.8	173.6	192.5	213.9	215.3	215.8	218.0	220.5	215.4	217.0	213.3	206.2	214.0	200.3	28.8	-
Growth capital costs	\$M	277.1	226.9	-	-	50.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining capital costs	\$M	1,062.0	116.1	72.2	90.4	107.8	67.4	75.4	57.9	54.6	50.5	63.1	60.8	45.1	53.1	69.3	44.5	33.6	0.3	-
Closure costs	\$M	40.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	40.0	-
All-in Sustaining Cost (“AISC”)	\$M	4,640.0	530.7	235.2	256.1	331.5	259.9	289.3	273.2	270.4	268.4	283.6	276.3	262.1	266.4	275.5	258.4	233.9	69.1	-
All-in Sustaining Cost (“AISC”)	US\$/oz	2,107	4,973	1,803	1,963	2,675	1,807	2,163	1,685	1,933	2,014	1,922	1,932	1,668	1,900	2,202	2,022	1,621	5,364	-

Table 22-3: Operating costs per tonne processed summary

Cost area	Unit	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18
Underground mining	\$/t	95.1	89.8	98.5	100.7	108.3	89.1	94.6	94.5	95.4	96.9	98.8	96.3	94.8	94.3	90.1	94.2	84.4	157.4	-
Processing - Mineralized material transportation	\$/t	0.7	15.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Processing – Toll milling	\$/t	2.7	61.5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Processing – On-site milling	\$/t	26.5	-	30.5	30.5	30.5	27.6	26.9	26.9	26.9	26.9	26.9	26.9	26.9	26.9	26.9	26.9	26.9	59.2	-
Waste and water management	\$/t	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	-
General and administrative	\$/t	10.1	13.4	12.3	12.3	12.7	10.0	9.3	9.3	9.3	9.3	9.3	9.3	9.3	9.3	9.3	9.3	9.3	31.6	-
Subtotal	\$/t	136.4	181.2	142.8	145.0	153.0	128.2	132.3	132.2	133.1	134.7	136.5	134.0	132.5	132.0	127.8	131.9	122.2	249.7	-
Selling costs	\$/t	0.5	0.5	0.6	0.6	0.6	0.5	0.4	0.5	0.4	0.4	0.5	0.5	0.5	0.5	0.4	0.4	0.5	0.6	-
Royalties	\$/t	4.4	4.9	5.4	5.4	4.9	4.2	4.0	4.6	4.4	4.3	4.0	2.8	5.7	3.9	3.6	4.2	5.4	-	-
Total	\$/t	141.3	186.7	148.8	151.0	158.5	132.9	136.7	137.3	137.9	139.3	140.9	137.3	138.7	136.4	131.8	136.4	128.1	250.2	-



22.8 Taxes

The Project is subject to different levels of taxation: federal corporate income tax, provincial (Québec) corporate income tax, mining duties, and carbon taxes. The federal and provincial corporate income tax rates currently applicable over the life of the Project are 15.0% and 11.5% of taxable corporate income, respectively. Québec mining duties range between 16% and 28% of mining-specific allowance-adjusted annual earnings, depending on yearly profitability margins. Opening balances for various tax attributes were considered based on Cartier Resources' recent tax returns.

It is estimated that a total of \$1,145.7 million in taxes will be paid over the life of the Project.

22.9 Economic Results

The main economic metrics used to evaluate the Project consist of the net undiscounted after-tax cash flow, the net discounted after-tax cash flow, the internal rate of return and the payback period. A 5% discount rate was used, as is common practice for gold projects.

A summary of the Project economic results is presented in Table 22-4 and the annual Project cash flows are presented in Table 22-5. Using the base case gold price, the total after-tax undiscounted cash flow over the Project life is \$1,867 million and after-tax NPV5% is \$1,001 million. The after-tax Project cash flow results in a 4.3-year payback period from the commencement of commercial operations with a pre-tax IRR of 35.5% and an after-tax IRR of 26.6%.



Table 22-4: Economic analysis highlights

	Unit	Base case	Spot case
Gold price	US\$/oz	3,600	4,300
Exchange rate	\$/US\$	1.38	1.38
Pre-tax free cash flow	\$ million	3,013	4,541
Pre-tax NPV5%	\$ million	1,679	2,630
Pre-tax IRR	%	35.5	51.0
Pre-tax payback period	Years	3.0	2.1
After-tax free cash flow	\$ million	1,867	2,772
After-tax NPV5%	\$ million	1,001	1,565
After-tax IRR	%	26.6	37.3
After-tax payback period	years	4.3	2.7





Table 22-5: Project cash flow summary

Description	Unit	Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18
Underground mining																						
Mineralized material mined	kt	23,145	66	240	891	1,082	1,135	1,200	1,200	1,584	1,584	1,584	1,584	1,584	1,584	1,584	1,584	1,584	1,585	1,443	48	-
Waste rock mined	kt	7,198	288	267	368	392	616	752	509	524	451	349	298	454	306	363	349	388	244	279	-	-
Processing																						
Mineralized material processed	kt	23,145	-	69	1,005	1,095	1,098	1,095	1,448	1,564	1,569	1,564	1,564	1,564	1,569	1,564	1,564	1,564	1,569	1,564	115	-
Gold grade processed	g/t	2.28	-	3.08	2.66	2.83	2.82	2.69	2.36	2.04	2.46	2.13	2.03	2.25	2.17	2.39	2.14	1.91	1.95	2.20	2.66	-
Gold in head feed	koz	1,698	-	7	86	99	99	95	110	103	124	107	102	113	110	120	108	96	98	111	10	-
Gold recovered	koz	1,610	-	6	78	95	95	90	105	97	118	102	97	107	104	114	102	91	93	105	9	-
Gold payable	koz	1,602	-	6	77	95	95	90	104	97	118	101	97	107	104	114	102	91	93	105	9	-
Cash flow																						
Net revenue	\$M	7,918	-	-	384	469	469	446	517	481	583	503	479	530	514	565	504	450	459	519	46	-
Royalties	\$M	- 102	-	-	-5	-6	-6	-5	-6	-6	-7	-7	-7	-6	-4	-9	-6	-6	-7	-9	-	-
Underground mining operating costs	\$M	2,194	-	-	90	108	111	119	129	148	148	149	152	154	151	148	147	141	148	132	18	-
Processing - Mineralized material transportation operating costs	\$M	15	-	-	15	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Processing - Toll milling operating costs	\$M	62	-	-	62	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Processing – on-site milling operating costs	\$M	611	-	-	-	33	33	33	40	42	42	42	42	42	42	42	42	42	42	42	7	-
Waste and water management operating costs	\$M	35	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	0	-
G&A operating costs	\$M	232	-	-	13	13	13	14	15	15	15	15	15	15	15	15	15	15	15	15	4	-
Total operating costs	\$M	3	-	-	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	-
Initial capital costs	\$M	276	87	189	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Growth capital costs	\$M	277	-	-	227	-	-	50	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining capital costs	\$M	1,062	-	-	116	72	90	108	67	75	58	55	50	63	61	45	53	69	44	34	0	-
Working capital variations	\$M	0	2	7	13	3	-1	-1	6	1	-0	2	-3	2	-0	-2	2	-3	0	-0	-27	-
Closure costs	\$M	40	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	40	-
Pre-tax cash flow	\$M	3,013	-89	-196	-160	232	215	116	252	192	311	231	215	246	239	305	236	178	201	286	4	-0
Taxes	\$M	- 1,146	-	-	-10	-37	-52	-44	-79	-60	-105	-77	-69	-90	-87	-109	-87	-65	-70	-102	-0	-
After-tax cash flow	\$M	1,867	-89	-196	-170	195	163	72	173	131	205	155	145	155	151	196	149	113	132	184	4	-0
Cumulative after-tax cash flow	\$M		-89	-285	-455	-260	-97	-25	148	279	484	639	784	940	1,091	1,287	1,436	1,548	1,680	1,864	1,867	1,867
Discounted pre-tax cash flow	\$M	1,679	-87	-182	-143	195	173	89	183	133	205	145	128	140	130	158	116	83	90	121	2	-0
Discounted after-tax cash flow	\$M	1,001	-87	-182	-152	164	131	55	126	91	135	97	87	89	82	101	73	53	59	78	2	-0



22.10 Sensitivity Analyses

The Project financial performance is most sensitive to the gold price and significantly less to the operating costs and capital expenditures. Table 22-6 summarizes the after-tax Net Present Value (5%), after-tax Internal Rate of Return and after-tax Payback Period sensitivities to the gold price, whereas Table 22-7 and Table 22-8 summarize sensitivities for capital costs and operating costs, respectively. Figure 22-3, Figure 22-4 and Figure 22-5 illustrate the sensitivities on the after-tax NPV 5%, IRR and Payback Period, respectively, to variations in the gold price, capital expenditures and operating expenses.

Table 22-6 : Gold price sensitivity analysis

Gold price (USD/oz)	After-Tax NPV5% (\$M)	After-Tax IRR	After-Tax Payback Period (years)
3,000	497	16.4%	6.2
3,600	1,001	26.6%	4.3
4,200	1,486	35.9%	2.8
4,800	1,962	44.9%	2.2
5,400	2,430	53.5%	1.9
6,000	2,896	62.0%	1.7

Table 22-7 : Capital costs sensitivity analysis

Variation	After-Tax NPV5% (\$M)	After-Tax IRR	After-Tax Payback Period (years)
-50%	1,406	58.5%	1.8
-40%	1,326	48.4%	2.1
-30%	1,246	41.0%	2.4
-20%	1,165	35.1%	2.8
-10%	1,083	30.5%	3.4
0%	1,001	26.6%	4.3
10%	918	23.4%	4.7
20%	833	20.5%	5.1
30%	747	18.1%	5.8
40%	663	15.9%	6.3
50%	579	14.1%	6.7



Table 22-8 : Operating costs sensitivity analysis

Variation	After-Tax NPV5% (\$M)	After-Tax IRR	After-Tax Payback Period (years)
-50%	1,555	37.3%	2.7
-40%	1,448	35.3%	2.8
-30%	1,340	33.2%	3.0
-20%	1,230	31.1%	3.4
-10%	1,116	28.9%	3.7
0%	1,001	26.6%	4.3
10%	883	24.3%	4.5
20%	762	21.8%	4.8
30%	639	19.3%	5.3
40%	516	16.8%	6.0
50%	385	14.0%	6.6

Figure 22-3 : After-tax Net Present Value (5%) sensitivity analysis

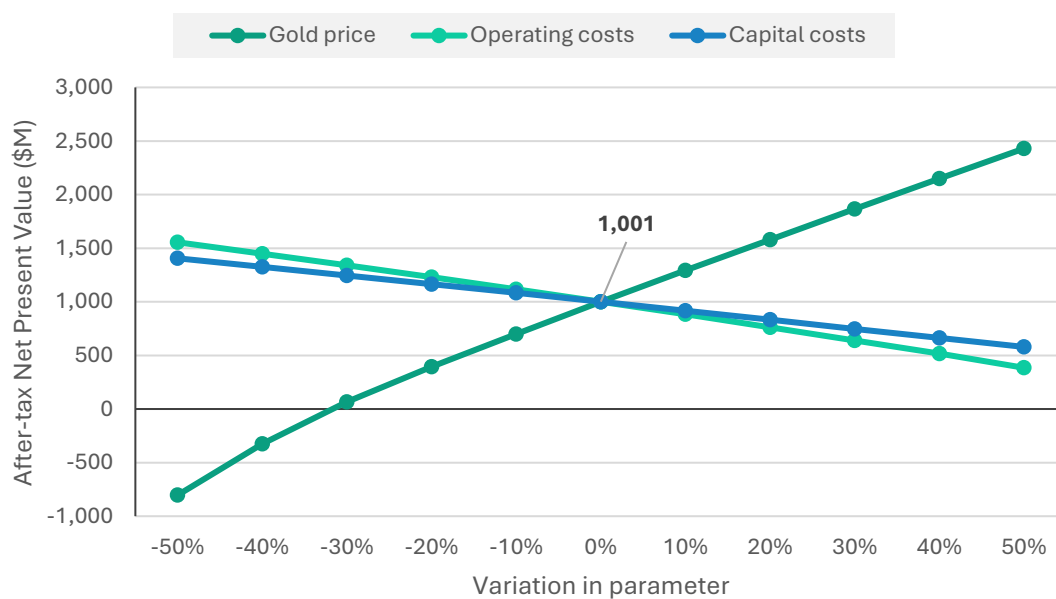


Figure 22-4 : After-tax Internal Rate of Return sensitivity analysis

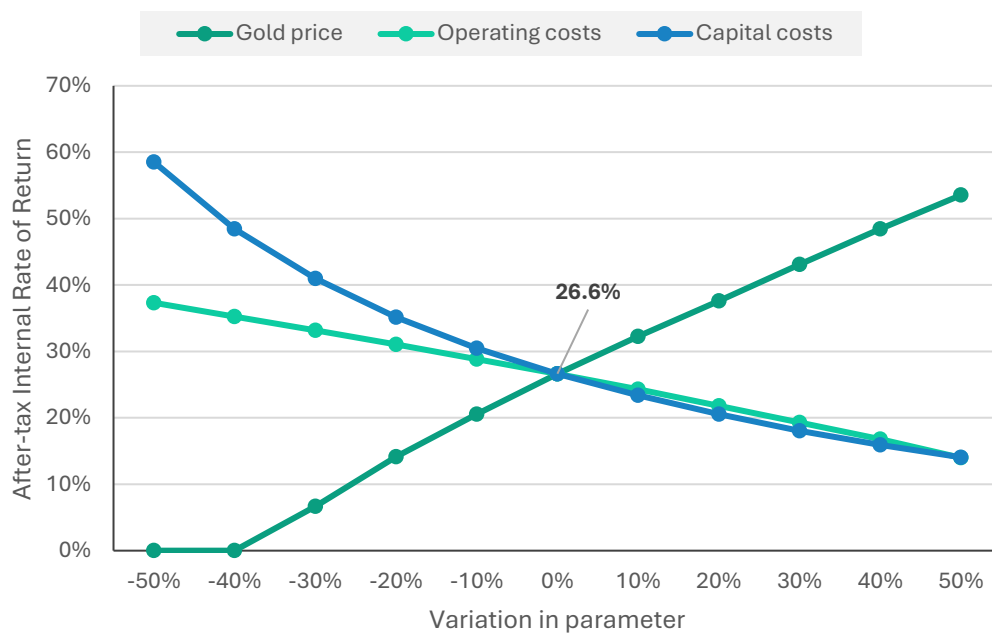
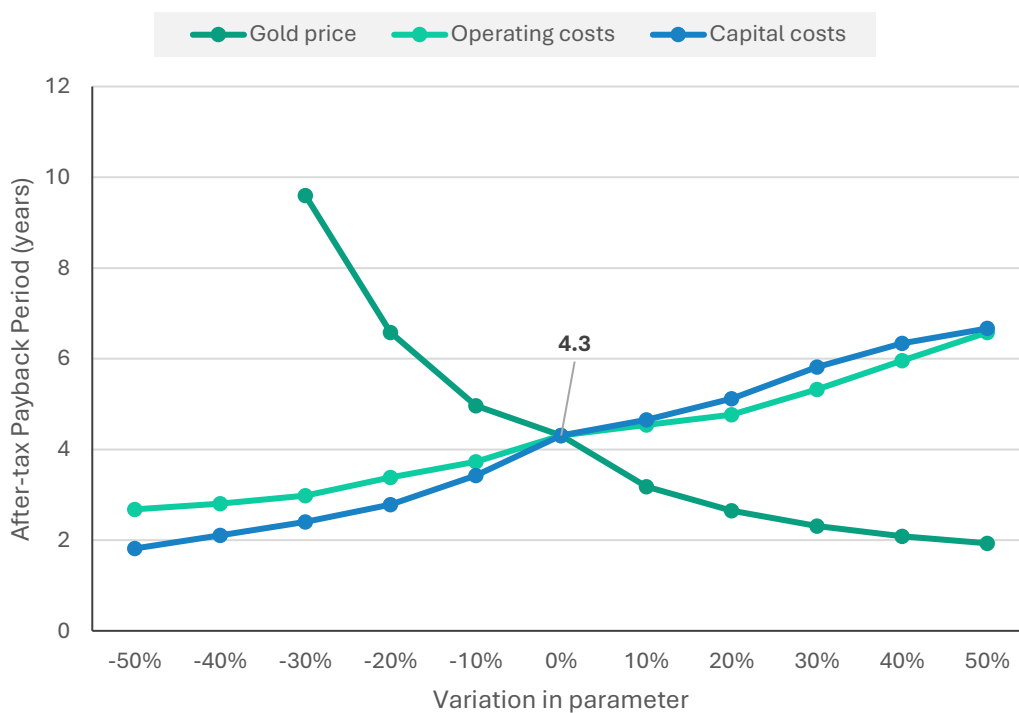


Figure 22-5 : After-tax Payback Period sensitivity analysis





23 ADJACENT PROPERTIES

The Project is located in the Val-d'Or mining camp. It is surrounded by several properties of varying sizes, some of which show signs of precious and base metals, but are only in the early stages of exploration. No adjacent properties present information deemed relevant to this Technical Report.



24 OTHER RELEVANT DATA AND INFORMATION

All relevant data and information regarding the Project have been disclosed under the relevant sections of this Report.

There is no other relevant data or information available that is necessary to make the current Report understandable and not misleading.



25 INTERPRETATION AND CONCLUSIONS

25.1 Overview

This Technical Report is prepared in accordance with the guidelines of the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101") and Form 43-101F1. The objective of this Report and the PEA is the evaluation of the potential technical feasibility and economic viability of the Project, notably the development of an underground mine thereat, including supporting infrastructure. This Report confirms the potential viability of the Project based on an underground mining operation that generates, on an after-tax basis, a NPV5% of \$1,001 million, a 4.3-year payback period and an IRR of 26.6%.

The QPs consider the PEA to be reliable, thorough, based on quality data, reasonable hypotheses, and parameters compliant with NI 43-101 requirements and CIM Definition Standards.

25.2 Geology and Mineralization

The understanding of the regional geology, lithological and structural controls of the mineralization at the Cadillac Project are sufficient to support the 2025 MRE.

25.3 Data Verification

The QP is of the opinion that the drilling protocols in place are adequate, and that the Project database is of good overall quality and suitable for mineral resource estimation.

25.4 Mineral Resources

The mineral resource estimate presented in this Report covers the Cadillac Project.

Leapfrog Geo™ and Edge™ v.2025.1.1 ("Leapfrog") was used to update the mineralized zones and to generate the drill hole intercepts for each solid. Leapfrog was used for the compositing, 3D block modelling, grade interpolation, and classification. Statistical studies were conducted using Excel, Leapfrog, and Supervisor. The pit optimization to develop the mineral resource-constraining pit shells was done using the pseudoflow algorithm in Deswik software. The stope optimization to develop the underground mineral resource was done using Deswik.SO software. The methodology for the mineral resource estimation involved the following steps:



- Database verification and validation;
- 3D modelling update of the mineralized zones;
- Drill hole intercepts and composite generation;
- Basic statistics;
- Capping;
- Geostatistical analysis including variography;
- Block modelling and grade interpolation;
- Block model validation;
- Mineral resource classification;
- Cut-off grade calculation;
- Pit shell optimization;
- DSO optimization;
- Preparation of the mineral resource statement.

The 2025 MRE is constrained within a pit shell developed and DSO shapes using appropriate cut-off grades. Table 25-1 presents the results of the MRE.

Table 25-1: Cadillac Project Mineral Resource Estimate

	Open Pit Resources			Underground Resources			Total		
	Cut-off Grade = 0.30g/t Au			Cut-off Grade = 1.00g/t Au					
	Tonnes (t)	Grade (Au g/t)	Gold (oz)	Tonnes (t)	Grade (Au g/t)	Gold (oz)	Tonnes (t)	Grade (Au g/t)	Gold (oz)
Measured	1,770,000	2.16	123,300	4,210,000	2.80	379,300	5,988,000	2.61	502,600
Indicated	1,730,000	1.52	84,500	2,240,000	2.51	180,600	3,965,000	2.08	265,200
M&I	3,500,000	1.84	207,800	6,450,000	2.70	559,900	9,953,000	2.40	767,800
Inferred	4,740,000	1.13	172,600	30,450,000	2.29	2,244,200	35,185,000	2.14	2,416,900

Notes to Table 25-1:

- The independent qualified person for the MRE, as defined by National Instrument (“NI”) 43-101 guidelines, is Pierre Luc Richard, P.Geo., of PLR Resources Inc. with contributions from Stephen Coates, P.Eng., of Evomine for cut-off values, open pit optimization solids and underground optimization solids, and Florent Baril, P.Eng., from Bumigeme Inc., for metallurgical parameters. The effective date of the MRE is November 11, 2025.
- These Mineral Resources are not mineral reserves as they have no demonstrated economic viability. No economic evaluation of these Mineral Resource has been produced. The quantity and grade of reported Inferred Resources in this MRE are uncertain in nature and there has been insufficient drilling to define these Inferred Resources as Indicated. However, it is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated category with continued drilling.
- The Qualified Persons are not aware of any known environmental, permitting, legal, title-related, taxation, socio-political,



marketing or other relevant issues that could materially affect the Mineral Resource Estimate.

4. Calculations used metric units (meters (m), tonnes (t), and g/t). Metal contents in the above table are presented in troy ounces (metric tonne x grade / 31.103475). Values were rounded, and any discrepancies in total amounts are due to rounding errors.
5. The Mineral Resource estimate follows the November 29, 2019, CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines.
6. Resources are presented as undiluted and in situ for the open-pit scenario and include internal dilution for the underground scenario and are considered to have reasonable prospects for economic extraction. The constraining pit shell was developed using overall pit slopes of 50 degrees. The pit optimization to develop the mineral resource-constraining pit shells was done using the pseudoflow algorithm in Deswik software. The stope optimization to develop the underground mineral resource was done using Deswik.SO software.
7. The MRE wireframe was prepared using Leapfrog Edge v.2025.1.1 and is based on 4,477 drill holes, totalling 535,360 meters drilled and 167,978 assays. The cut-off date for the drill hole database was February 17, 2025.
8. Composites of 1.0 metre were created inside the mineralization domains. High-grade capping was done on the composited assay data. Based on individual statistical study for each zone, composites were capped between 5.0 g/t Au and 110.0 g/t Au for the high-grade zones, and between 1.0 g/t Au and 5.0 g/t Au for the low-grade zones.
9. Pit constrained Mineral Resources for the base case are reported at a cut-off grade of 0.30 g/t Au; DSO-constrained Mineral Resources for the base case are reported at a cut-off grade of 1.00 g/t Au and include internal dilution (must-take). The cut-off grades will be re-evaluated in light of future prevailing market conditions and costs.
10. Specific gravity values were estimated using data available in the drill hole database. Density values between 2.80 and 2.88 were applied to the model for hard rock and 2.00 for overburden.
11. Grade model resource estimation was calculated from drill hole data using an Ordinary Kriging interpolation method in a sub-blocked model using blocks measuring 5 m x 5 m x 5 m in size and sub-blocks down to 0.625m x 0.625m x 0.625m. Ordinary kriging (OK), inverse square distance (ID2), Nearest neighbour (NN) interpolation methods were tested, resulting in no material difference in the Mineral Resource Estimates.
12. The Measured, Indicated and Inferred Mineral Resource categories are constrained to areas where drill spacing is less than 12.5m, 25m and 100 metres respectively and show reasonable geological and grade continuity. An additional requirement for the Measured category is the close proximity of underground infrastructure. Cookie cutters were used to define categories based on the above parameters. Based on historical mining and geological knowledge of the deposit, drill spacing was increased up to 140m in the shoot direction to define inferred resources for some zones.

25.5 Exploration Potential

After reviewing all pertinent information, including the MRE, the QP concluded the following:

- The potential is high for adding underground mineral resources to the Cadillac Project by extending 3D modelling at depth and laterally.
- The potential to upgrade Inferred Mineral Resources to the Indicated category with additional drilling is high.
- The exploration potential remains high at the Project scale, justifying further geological compilation and continuing exploration target generation programs.



A Conceptual Exploration Target, with underground potential, was identified during the preparation of the MRE. This conceptual Exploration Target is integrated into the model used for the MRE, with the aim of facilitating future targeting and drill hole planning.

The assessment of the target for further exploration was completed by PLR Resources Inc. with contribution from Evomine. The estimation of the potential quantity and grade of the exploration target was based on the same drill hole database used for the Mineral Resource Estimate. With the available drilling information, PLR developed conceptual gold mineralization volumes. The original core samples were composited, and the composited gold assays were capped (similarly to the Mineral Resource Estimate) after evaluating the statistical distributions on probability plots. The gold values were interpolated into a three-dimensional block model using Ordinary Kriging. To estimate a tonnage, PLR used the same specific gravity values used for the Mineral Resource Estimate.

DSO stopes were run to constrain the Exploration Target.

The Conceptual Exploration Target is estimated to be of 8 to 12 million tonnes of mineralization grading between 2.2 to 2.8 g/t Au, representing 600,000 to 1,100,000 ounces of gold.

Please note the following disclosure warnings in respect to this exploration target:

- An Exploration Target is not a National Instrument 43-101 compliant resource or reserve.
- The Exploration Target is confirmed only as a target for further exploration.
- Potential quantity and grades are conceptual in nature only.
- There has not been sufficient drilling to define any mineral resource on this Exploration Target; drilling intercepts crosscut the Exploration Target but drill spacing is too scarce to classify these blocks as Inferred Mineral Resources.
- There is no certainty that further drilling will result in the target being delineated as a mineral resource.

25.6 Metallurgical Testing and Mineral Processing

A single metallurgical testwork program has been completed on the Cadillac Project. It was carried out by ALS Metallurgy under job number KM7798 between December 2025 and March 2026, on nine composites prepared from the three deposits considered in this study — Chimo, East Chimo and West Nordeau. Forty-four tests were completed, covering grindability, gravity separation, whole-rock cyanidation at three grind sizes per composite, cyanidation of gravity tailings, and cyanide destruction.



The results support the following conclusions:

- Mineralogy is consistent between the three deposits, with moderate sulphide contents and no deleterious minerals identified. Pyrrhotite is more abundant in Chimo and East Chimo, and arsenopyrite in West Nordeau. The variability composites show limited intra-deposit variability, which supports the representativeness of the master composites;
- Comminution characteristics are comparable across the three deposits, with Bond ball mill work indices of 13.1 kWh/t to 14.3 kWh/t and SCSE values of 11.3 kWh/t to 13.1 kWh/t. These values indicate moderate to high competency relative to the JKTech database, but all three deposits are expected to respond comparably in a conventional crushing and grinding circuit;
- Gold is amenable to whole-rock cyanidation. Under standard conditions, recovery ranges from 90.1% to 98.0% at a K80 of 48 µm to 56 µm, and decreases to 77.6% to 87.7% at a K80 above 120 µm, confirming a marked sensitivity to grind size;
- Lead nitrate addition combined with pre-aeration accelerates gold dissolution and, at the finest grind size, reduces average cyanide consumption from 0.92 kg/t to between 0.19 kg/t and 0.32 kg/t, with gold recovery ranging from 93.4% to 97.8%. This condition has been retained in the conceptual design basis;
- A gravity-recoverable gold fraction of 43.9% to 61.4% is present in all three deposits at the design grind size, but cyanidation of the gravity tailings returned an equivalent overall recovery, so no recovery credit has been taken for gravity separation;
- Cyanide destruction by the SO₂/air process was effective but required a higher than usual SO₂ dosage to reduce the CN_{wad} below 2 ppm. This is probably a result of the presence of a significant pyrrhotite content. A dosage of 8 gSO₂/gCN_{wad} is then required instead of a value of 5 that usually work for mill feed that contains less sulphide.

A recovery equation based on grind size, feed gold grade and feed sulphur grade was established from the leach results and is used to project gold recovery over the mine plan. The testwork provides an adequate basis for projecting the metallurgical performance of cyanide leach processing and estimating the process operating cost at the preliminary economic assessment level. Further testwork will be required to advance the project beyond this level.

The proposed recovery flowsheet uses conventional, proven unit operations and is supported by the available metallurgical testwork. The plant is designed to process 3,000 t/d in Phase 1 and 4,300 t/d



in Phase 2 through crushing, SAG and ball milling, gravity recovery, pre-aeration, CIL leaching, carbon elution and regeneration, electrowinning, smelting, and cyanide detoxification. The design targets overall gold recoveries of 95.0% in Phase 1 and 94.8% in Phase 2, with a 50 µm final grind maintained to support these recoveries.

Overall, the selected process route is appropriate for the mineralogical characteristics of the Chimo, East Chimo, and West Nordeau deposits and provides a practical basis for phased project development.

The mill design is based on preliminary economic evaluation requirements. Accordingly, only major equipment has been sized for cost estimation using a high-level process flow diagram. Further studies will include trade-off analyses of grinding circuit configurations and alternative grinding technologies. If further metallurgical testwork indicates that sulphide flotation with concentrate regrind is potentially economic, this process will need to be added. The gold leach and recovery process will also be optimized by evaluating different carbon management approaches. A detailed process flow diagram and high-level P&IDs will also be required to generate more detailed equipment and instrument lists, thereby reducing the global contingency.

25.7 Mining

The Cadillac Project is planned as a mechanized underground mining operation using longitudinal longhole open stoping (LHOS), with a combination of cemented rockfill, uncemented rockfill and paste fill for stope backfilling. This mining method was selected based on the geometry of the mineralized zones, historical mining experience at the historic Chimo Mine, and the anticipated ground conditions. It provides a mechanized, non-entry mining approach intended to support productive and cost-effective extraction.

The mine plan encompasses six mineralized domains: Main–Chimo, Main–East, Nordeau, Rapid, Contact and Portal. These domains will be accessed through five primary surface portals and dedicated ramp systems, with mining areas developed and extracted sequentially from the bottom upward. The planned stopes have a nominal vertical height of 25 m and a strike length of 17.5 m, with a minimum mining width of 2.0 m. Mining dilution allowances are based on equivalent linear overbreak/slough (ELOS) factors of 0.5 m for the hanging wall and 0.2 m for the footwall. A 30 m buffer is maintained around historical mine workings to mitigate potential ground stability risks.

The underground mine schedule includes a 24-month pre-production period dedicated to portal construction, ramp and lateral development, ventilation infrastructure, and initial stope



preparation. Following a three-month production ramp-up, the operation is scheduled to achieve a nominal production rate of 3,000 tonnes per day (tpd) during the first four years of commercial operations, increasing to 4,300 tpd in Year 5 and remaining at this rate for most of the remaining mine life. The overall project schedule comprises 19 years, including pre-production and 17 years of commercial operations. Approximately 23.1 million tonnes of mineralized material are scheduled to be mined at an average diluted gold grade of 2.28 g/t.

The mine design requires approximately 234 km of lateral underground development and 6.4 km of vertical development. Mining will be performed by a contractor using a mechanized fleet that includes production and development drills, load-haul-dump (LHD) units, underground haul trucks, rockbolters and auxiliary equipment. Mineralized material will be hauled directly to surface and stockpiled near the portals before delivery to the processing facility. Development waste rock will be transported to designated surface waste rock management facilities or reused underground as backfill, where appropriate.

Backfilling is an integral component of the mining sequence and ground stability strategy. The use of paste fill is expected to allow a portion of the processing tailings to be returned underground, reducing the volume of tailings requiring surface storage and potentially limiting additional waste rock requirements. Ventilation will be provided through a staged system comprising temporary ventilation during initial development and a permanent network of fresh-air raises, surface fans and booster fans. Underground services will also include dewatering, compressed air, communications, fuel distribution and emergency infrastructure.

The mining design and production schedule are preliminary and rely on historic geomechanical information, empirical design methods and experience from comparable operations. No new site-specific geomechanical or hydrogeological investigations were completed for this assessment. Further geotechnical drilling, rock mass characterization, hydrogeological studies, and validation of stope stability, ground support, crown pillar dimensions and backfill strength will therefore be required during subsequent engineering studies to confirm and optimize the mine design.

25.8 Environmental Studies, Permitting and Social or Community Impact

The construction, operation, and closure of the mine are subjected to several laws and regulations at the federal, provincial, and municipal levels. The Cadillac project is not subject to the federal



environmental assessment procedure with a production capacity below 5,000 t/d. At the provincial level, all work required for the operation of a new mine are subject to the impact assessment procedure. The Cadillac project will be subject to the procedure for impact assessment of development projects in meridional Quebec with an eventual review by the BAPE. It is recommended to start baseline studies and consultations with stakeholders early in the development of the project. Moreover, the Mining Act provides a framework for the mining lease, rehabilitation and restoration plan, and financial guarantee. The closure plan must be approved by the MRNF before the issue of the mining lease, and a financial guarantee to fully implement the closure plan must be provided in the first two years following its approval. Estimated closure costs of approximately C\$40 millions have been benchmarked against recent mining projects in similar jurisdictions.

Since 2026, Cartier has initiated a series of environmental studies with Stantec to understand the baseline environmental conditions of the project site. Major components of the physical, biological and human environments have been described in an initial desk review study. Environmental baseline studies will start in the next phase of the project. Geochemical characterization of waste rock and mineralized material is currently ongoing.

As part of the development of the Cadillac project, several information and consultation meetings with stakeholders and First Nation authorities are required. An engagement process will be implemented by Cartier to meet various stakeholders concerned by the Cadillac project, including municipal and regional authorities, development economic organizations, environmental groups, nearby cottagers and land users. Meetings will also be held with representatives of the Anishnabe Nation Council of Lac-Simon.

25.9 Capital and Operating Costs

Life-of-mine project capital costs are estimated to total \$1,685.1 million gross of pre-production revenue (\$1,654.9 million net of pre-production revenue), split in the following the following categories:

- Initial Capital Costs – This category includes all costs to develop the property and does not include a process plant, as the project starts with a toll mining operation for the first year of production. Initial capital costs total \$306.3 million (including \$37.9 million for contingency but gross of \$30.6 million in pre-production revenue), which will be expended over a 24-month period of construction, pre-production and commissioning.



- Growth capital costs – This category includes all costs related to the construction of process plant expended during the first year of production, as well as the expansion of this plant during the fourth year of production. Growth capital costs are estimated to be \$277.1 million, inclusive of a \$37.8 million contingency.
- Sustaining Capital Costs – This category includes all costs related to the acquisition, replacement, or major overhaul of assets required to sustain operations, as well as underground development and infrastructure during the production phase of the Project. Sustaining capital costs are estimated to be \$1,062.0 million, inclusive of a \$138.5 million contingency.
- Working Capital – This category consists of costs related to building up an inventory. It is planned to maintain a supplies inventory equivalent to 45 days of operating expenses, as well as a finished goods inventory equivalent to 7 days of revenue. Additionally, it is planned to pay the various operational suppliers within 30 days, and 7 days of accounts receivable are considered. Though the total net changes in working capital over the life of mine is null, the maximum amount of working capital for the Project is \$34.9 million.
- Closure Costs – This category includes all costs related to the closure, reclamation, and ongoing monitoring of the mine after operations. Closure costs are estimated at \$40.0 million.

The total capital costs are summarized in Table 25-2.



Table 25-2 : Capital Costs Summary

Cost Area	Initial Capital Costs (\$M)	Growth Capital Costs (\$M)	Sustaining Capital Costs (\$M)	Total Capital Costs (\$M)
Infrastructure	40.9	-	52.5	93.4
Electrical and communication	27.9	-	8.1	36.0
Waste and water management	9.2	-	58.7	68.0
Mine	91.3	-	720.2	811.5
Process plant	9.0	230.2	-	239.2
Indirect costs	29.8	9.0	84.0	122.7
Contingency	37.9	37.8	138.5	214.2
Capitalized operating costs	60.0	-	-	60.0
Closure	-	-	40.0	40.0
Total	306.0	277.1	1,102.0	1,685.1
Pre-production revenue	-30.2	-	-	-30.2
Total (net of pre-production revenue)	275.8	27..1	1,102.0	1,654.9

Operating costs are summarized in Table 25-3. The operating costs include mining, processing, waste and water management and general services and administrative (“G&A”) costs. The average life of mine operating cost is US\$1,481/oz of gold payable, including selling and royalty costs. The average life of mine all-in sustaining cost (“AISC”) is US\$2,107/oz of gold payable.



Table 25-3 : Operating Costs Summary

Cost Area	Total Cost (\$ M)	Unit Cost		Unit Cost (US\$/oz payable)
Underground mining	2,193.7	95.1	\$/tonne processed	996.0
Off-site transport	15.1	0.7	\$/tonne processed	6.8
Off-site processing	61.8	2.7	\$/tonne processed	28.1
On-site processing	610.5	26.5	\$/tonne processed	277.2
Waste and water management	34.6	1.5	\$/tonne processed	15.7
General and administrative	232.5	10.1	\$/tonne processed	105.6
Subtotal	3,148.2	136.4	\$/tonne processed	1,429.4
Selling costs	11.1	0.5	\$/tonne processed	5.0
Royalty (1.5%)	101.7	4.4	\$/tonne processed	46.2
Total	3,260.9	141.3	\$/tonne processed	1,480.6
Growth capital	277.1	12.0	\$/tonne processed	125.8
Sustaining capital (incl. closure costs)	1,102.0	47.8	\$/tonne processed	500.4
All-in Sustaining Costs ("AISC")	4,640.0	201.1	\$/tonne processed	2,107.0

The total workforce for the Cadillac Project, which includes personnel for the mining operation, transport to the toll milling facility, and general and administrative activities is estimated at 515.

25.10 Economic Analysis

The economic and financial evaluation presented in this Technical Report utilizes a discounted cash flow method, both on a pre-tax and after-tax basis. The gold price used in the evaluation was US\$3,600 per troy ounce, as discussed in Item 19 of this Technical Report. The financial model provides results in terms of NPV, payback period, and IRR for the Cadillac project. The economic analysis is conducted in real terms, without considering inflation factors, using Q3 2026 Canadian dollars. The analysis does not take into account project financing.

The economic model estimates cash flows on an annual basis for the life of the Cadillac project, based on the level of engineering and design appropriate for a PEA.

Cash flow projections for the life of the mine are based on sales revenue, operating costs, capital expenditures and other cost estimates. Capital costs are estimated in 5 categories: initial capital,



growth capital, sustaining capital, closure and working capital. Operating cost estimates include labour, reagents, maintenance, supplies, services, fuel, and power. Other costs, such as royalties, depreciation, and taxes, are estimated based on the current project and processing plans.

The economic results are calculated from the start of the initial capital expenditures.

A summary of the Project economic results is presented in Table 25-4. The total after-tax undiscounted cash flow over the Project life is \$1,867 million and after-tax NPV5% is \$1,001 million. The after-tax Project cash flow results in a 4.3-year payback period from the commencement of commercial operations with an after-tax IRR of 26.6%.

Table 25-4: Economic analysis highlights

	Unit	Base case	Spot case
Gold price	US\$/oz	3,600	4,300
Exchange rate	\$/US\$	1.38	1.38
Pre-tax free cash flow	\$ million	3,013	4,541
Pre-tax NPV5%	\$ million	1,679	2,630
Pre-tax IRR	%	35.5	51.0
Pre-tax payback period	Years	3.0	2.1
After-tax free cash flow	\$ million	1,867	2,772
After-tax NPV5%	\$ million	1,001	1,565
After-tax IRR	%	26.6	37.3
After-tax payback period	years	4.3	2.7

25.11 Opportunities

There are several opportunities that could improve the economics, timing, and/or permitting potential of the Project. The key opportunities that have been identified at this time are as follows:

- Exploration has to potential for discovery of extensions to existing zones or new zones that could be added to the Project;
- Implementation of appropriate mitigation measures to reduce environmental and social impacts;
- Consideration of the stakeholders concerns in the project development to ensure a better social acceptance;



- Reprocessing of the tailings from the historic Chimo mine;
- Enhancement of metallurgical and mineralogical knowledge could improve recovery assumptions;
- Development of a scenario using current M&I Resources to potentially accelerate the timeline towards the delivery of a Pre-Feasibility or Feasibility Study;
- Increase in the gold price.

25.12 Risks

The key risks identified at this time for the Project are as follows:

- The mineralization encountered might present differences from the current geological model, leading to lower realized tonnages and/or gold grades versus that which are expected;
- The planned daily mining production rate may prove difficult to achieve due to geological continuity issues, geotechnical issues, possible interaction of equipment, automation constraints, and other potential slowdowns resulting in a longer mining cycle time;
- Ground conditions could be different than expected and influence stope dimensions, pillar recovery, ground support, development and mining sequence, and dilution and recovery;
- Greater water inflow than anticipated leading to an increase in water pumping and treatment capital and operational costs;
- Metallurgical recovery assumptions remain subject to variability associated with mineralized material mineralogy, grade distribution, and scale-up performance; deviations from projected recoveries could materially impact process plant performance, operating costs, and overall project economics;
- The Project's reliance on a third-party toll milling arrangement introduces risks related to the successful negotiation and execution of definitive agreements, including availability of mill capacity, commercial terms, permitting compatibility, metallurgical suitability, scheduling constraints, and potential interruptions to processing access that could adversely affect production schedules and project economics;
- The Project is subject to permitting and regulatory approval risks, including potential delays in obtaining, maintaining, or amending required permits and authorizations; changes in applicable environmental or mining regulations, stakeholder concerns, or additional



information requests from regulatory agencies could impact the Project development schedule, operating conditions, and overall economics. Limited workforce availability;

- Limited goods and services availability;
- Inflationary pressure, notably due to increased fuel pricing;
- Downturn in the gold price.



26 RECOMMENDATIONS

The results of the Preliminary Economic Assessment (“PEA”) indicate that the Project has the potential to be economically attractive under the assumptions and parameters considered in this study. The economic analysis presents positive valuation metrics and supports advancing the Project to the next stage of technical evaluation. However, the assessment is preliminary in nature, and significant technical investigations, engineering studies, and further evaluation of key risks are required to improve the level of confidence in the Project's development concept and economic performance.

It is therefore recommended that the Project advance to a structured program of technical work aimed at validating the principal assumptions of the PEA, reducing key technical and economic uncertainties, and establishing a sound basis for a future Pre-Feasibility Study (“PFS”). The recommended work program should include, but not necessarily be limited to, the following activities.

Mineral Resources and Drilling

- Undertake additional infill drilling within the areas that contribute most significantly to the proposed mine plan, with the objective of improving confidence in the Mineral Resource Estimate and supporting the potential conversion of Inferred Mineral Resources to Indicated Mineral Resources, where justified.
- Conduct additional geological and structural investigations to refine the understanding of mineralization geometry, continuity, and relevant geological controls.
- Update the geological model and Mineral Resource estimate as warranted by the results of the additional drilling and supporting investigations.

Geotechnical and Mine Design

- Conduct a geotechnical investigation program, including oriented core drilling, rock mass characterization, structural data collection, and laboratory testing, to support the development of geotechnical domains and appropriate rock mass design parameters.
- Complete preliminary underground mine geotechnical analyses to establish appropriate excavation dimensions, ground support requirements, stope design parameters, and mine stability criteria.
- Refine the mine design, production schedule, development requirements, and mining method assumptions based on the updated geological model and geotechnical findings.



- Evaluate the implications of the revised mine design on mining productivity, dilution, mineralized material loss, mine sequencing, and operating costs.

Hydrogeology and Water Management

- Undertake a hydrogeological investigation program to characterize groundwater conditions, hydraulic properties, potential groundwater inflows, and the principal hydrogeological controls affecting mine development and operations.
- Develop a preliminary groundwater model, as appropriate, to support estimates of mine inflows, dewatering requirements, and potential effects on surrounding groundwater and surface water systems.
- Refine the mine water management concept, including water collection, pumping, treatment, storage, discharge, and/or reinjection requirements, as applicable to the Project.

Metallurgy and Mineral Processing

- Complete additional metallurgical testwork to confirm and refine the assumed metallurgical recoveries and assess variability in mineralized material characteristics across the proposed mine plan.
- Investigate potential impacts of mineralogical variability, deleterious elements, and processing conditions on concentrate quality, recoveries, and operating costs.
- Refine the process flowsheet, plant design criteria, and associated capital and operating cost estimates based on the results of the additional testwork.
- Engage in discussions with third party mill operators to establish a toll milling agreement.

Infrastructure, Waste Management, and Environmental Studies

- Advance the engineering of the principal site infrastructure, including access roads, electrical supply, water management facilities, and other facilities required to support the proposed mining and processing operations.
- Complete preliminary geotechnical and geochemical characterization of waste rock and other relevant mine materials to support the design of waste management facilities and the assessment of potential environmental liabilities.
- Advance the conceptual designs for tailings and waste rock management facilities, including site selection, capacity requirements, foundation conditions, and water management considerations, as applicable.
- Initiate or continue baseline environmental studies and permitting-related investigations to



identify potential environmental constraints, assess the principal project impacts, and support future regulatory submissions.

- Engage with relevant stakeholders and First Nations communities, as appropriate, to identify potential concerns and incorporate applicable considerations in Project development.

Engineering and Economic Evaluation

- Advance engineering and design activities to an appropriate level of detail for a PFS, incorporating the results of the recommended technical investigations.
- Update capital and operating cost estimates using refined engineering quantities, equipment requirements, productivity assumptions, and current pricing information.
- Update the production schedule and economic model to reflect the revised technical assumptions, cost estimates, metallurgical parameters, and any material changes to the proposed development concept.
- Conduct sensitivity analyses and risk assessments to identify the principal factors affecting Project economics and determine which uncertainties warrant further investigation.
- Evaluate opportunities to optimize the mine plan, infrastructure configuration, processing strategy, and overall development approach, with emphasis on improving economic performance while managing technical and execution risks.

The recommended work should be prioritized according to its potential to affect the Project's economic performance and the level of uncertainty associated with the relevant assumptions. In particular, the drilling, geotechnical, hydrogeological, and metallurgical programs should be designed to address the uncertainties that have the greatest potential to influence the mine plan, capital and operating costs, and economic valuation.

Subject to the results of these investigations and confirmation that the principal technical and economic assumptions remain supportable, the Project should then be advanced to a Pre-Feasibility Study. The PFS should establish a more detailed and integrated mine development plan, infrastructure design, production schedule, cost estimate, and economic evaluation, while further assessing the principal technical, environmental, permitting, and execution risks. Table 26-1 highlights the proposed budget for the work program.



Table 26-1: Work program budget

Item	Budget
Expansion and definition drilling	\$31,250,000
Exploration drilling	\$3,750,000
Geotechnical investigation and analysis	\$750,000
Hydrogeological investigation and analysis	\$750,000
Metallurgical and mineralogical testwork	\$750,000
Geochemical characterization of material	\$250,000
Environmental baseline	\$2,000,000
Stakeholder engagement	\$100,000
Pre-Feasibility Study including updated Mineral Resource Estimate	\$2,000,000
Total	\$41,600,000

Although the PEA presents positive preliminary economic results, additional work is required before the Project's technical feasibility and economic viability can be established with greater confidence. The results of the PEA should therefore be considered an initial assessment of the Project's potential rather than a definitive basis for a development decision.





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