NI 43-101 Feasibility Study: Troilus Gold – Copper Project
Québec Canada

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IMPORTANT NOTICE

Contents

Tables

Figures

Appendices

Forward Looking Statements

This Technical Report, including the economics analysis, contains statements or information that constitute forward-looking information (forward-looking statements) within the meaning of applicable Canadian securities laws. Forward-looking statements include, but are not limited to, statements regarding the results of the FS, including, without limitation various project economics, financial and operational parameters such as the timing and amount of future production from the Project, expectations with respect to the IRR, NPV, payback and costs of the Project, anticipated mining and processing methods of the Project; proposed infrastructures, anticipated mine life of the Project, expected recoveries and grades, timing of future studies and development plans, the timing and progress of the Federal and Provincial permitting processes; the development potential and timetable of the project; the estimation of mineral resources and reserves; realization of mineral resource and reserve estimates; the timing, success and amount of estimated future exploration; costs of future activities; capital and operating expenditures; and success of exploration activities. Generally, forward-looking statements can be identified by the use of forward-looking terminology such as “plans”, “expects” or “does not expect”, “is expected”, “budget”, “scheduled”, “estimates”, “forecasts”, “intends”, “continue”, “anticipates” or “does not anticipate”, or “believes”, or variations of such words and phrases or statements that certain actions, events or results “may”, “could”, “would”, “will”, “might” or “will be taken”, “occur” or “be achieved”. Troilus Gold Corp. Forward-looking statements are made based upon certain assumptions and other important facts that, if untrue, could cause the actual results, performance, or achievements of Troilus to be materially different from future results, performances or achievements expressed or implied by such statements. Such statements and information are based on numerous assumptions some of which are discussed in this Technical Report. Forward-looking statements are subject to known and unknown risks, uncertainties and other important factors that may cause the actual results, level of activity, performance or achievements of Troilus to be materially different from those expressed or implied by such forward-looking statements, including but not limited to: there being no assurance that the exploration program or programs of Troilus will result in expanded mineral resources; risks and uncertainties inherent to mineral resource and reserve estimates; the high degree of uncertainties inherent to feasibility studies and economic analysis which are based to a significant extent on various assumptions; variations in gold prices and other metals; exchange rate fluctuations; variations in cost of supplies, labour rates and consumable and equipment costs; receipt of necessary approvals; availability of financing for project development; uncertainties and risks with respect to developing mining projects; general business, economic, competitive, political and social uncertainties; future gold and other metal prices; accidents, labour disputes and shortages; environmental and other risks of the mining industry, including without limitation, risks and uncertainties discussed in the Company's latest Annual Information Form and other continuous disclosure documents of the Company available under the Company's profile at www.sedarplus.ca. There may be other factors that cause results not to be as anticipated, estimated or intended. There can be no assurance that such statements will prove to be accurate, as actual results and future events could differ materially from those anticipated in such statements. Accordingly, readers should not place undue reliance on forward-looking statements.

Non-IFRS Financial Measures

This Technical Report includes certain non-IFRS financial measures or ratios, such as All-In Sustaining Cost and Cash Costs, which are not measures recognized under IFRS and do not have a standardized meaning prescribed by IFRS. As a result, these measures may not be comparable to similar measures reported by other corporations. Each of these measures used are intended to provide additional information to the user and should not be considered in isolation or as a substitute for measures prepared in accordance with IFRS.

As construction and operation of the Project are at the study stage, there are no historical non-IFRS financial measures nor historical comparable measures under IFRS, and therefore the foregoing prospective non-IFRS financial measures or ratios may not be reconciled to the nearest comparable measures under IFRS.

1 SUMMARY

1.1 Introduction

Troilus Gold Corp. (Troilus) is a Canadian exploration company with its corporate office located in Montreal, Canada. The Troilus Gold Property (Property) is divided into two projects: the Troilus Gold - Copper Project (Project), and the Troilus Frotêt Project. The mineral rights to the Property cover a total area of approximately 44,124.88 ha. Of this total area, 7,242 ha is 50% owned by Troilus and 50% owned by Argonaut Gold Inc. through a joint venture (JV) agreement, with the remainder of the mineral rights being 100% held by Troilus.

The Troilus Mine was originally an open pit operation producing gold, copper, and silver continuously from November 1996 to April 2009. The Troilus Mine produced over 2 million oz of gold and approximately 70,000 t of copper. After the mine ceased production in 2009, the 20,000 t/d mill processed low grade stockpiles until the end of June 2010. Following this, the mill was sold and shipped to Mexico and the main camp facilities were dismantled in late 2010. A significant amount of site infrastructure was left in place after the mine closure and disposition of some of the key assets.

This Technical Report (Report) was prepared on behalf of Troilus, by AGP Mining Consultants Inc. (AGP). The purpose of the Report is to present the results of the feasibility study and the Mineral Reserves estimate of the four principal deposits of the Project. This Report was prepared in compliance with the Canadian disclosure requirements as provided in National Instrument 43-101 (NI 43-101) and in accordance with the requirements of Form 43-101 F1.

The effective date of the Mineral Reserves for this report is 15 January 2024.

1.2 Location and Ownership

The Project is located in central Québec and is situated approximately 170 km north of Chibougamau. The mineral rights to the Property cover a total area of approximately 44,124.88 ha. Of this total area, 7,242 ha is 50% owned by Troilus and 50% owned by Argonaut Gold through a joint venture (JV) agreement, with the remainder of the mineral rights being 100% held by Troilus.

The mineral rights to the Project are comprised of a single Mining Lease (Bail Minier) and 293 mineral claims (Titres Miniers), totalling 16,185.09 ha. The mineral rights to the Troilus Frotêt Project is comprised of 520 mineral claims, totalling 27,939.79 ha. All mineral rights are in good standing.

1.3 Accessibility, Local Resources, Infrastructure and Climate

The Project is easily accessible by road from Chibougamau, Québec, along Highway 167 and the Route du Nord, which begins approximately 18 km northeast of Chibougamau. The Route du Nord is a maintained all-weather dirt road and is open year-round. The Project site follows the Route du Nord for approximately 108 km, to the turn off east along the Troilus Mine Road for roughly 44 km. The drive from Chibougamau is typically 2 hours.

The region where the Property is situated has a Continental Subarctic climate characterized by long cold winters and short mild summers. Exploration and mining activities may be carried out all year round.

The nearest town to the Property is Mistissini, a Cree community located approximately 90 km southeast of the site. Chibougamau, with a population approximately 7,500 (est. 2016), is the largest town in Nord-du-Québec, and offers most services, supplies and fuel required for the Project. Chibougamau is a well-established mining town and has a well-developed local infrastructure, services, and a mining industry workforce. The Property is connected to the provincial hydroelectric grid via a 137 km 161 kV power line. Water on the Property is abundant and available for exploration activities. Troilus maintains an 80-person camp (accommodation and kitchen), including but not limited to: permanent and semi-permanent buildings for: administration, exploration, core logging and sampling; garage; electrical transformer station; tailings treatment plant, etc.

1.4 History

Prior to 1985, the Project area was subject to regional exploration by Falconbridge Ltd. (now Glencore) and Selco Mining Corp. The Government of Québec also conducted an airborne survey over a large area of the eastern portion of the Frotêt-Evans belt.

In 1987, mineralization in the Project area was discovered by Kerr Addison and by 1993 a positive feasibility study was published. The mine started commercial production in October 1996 and operated continuously up to April 2009 and the mill continued to process stockpile material up to June 29, 2010.

From 1995 to 2010, approximately 69.6 million tonnes (Mt) averaging 1.00 g/t Au and 0.10% Cu of ore was mined and 7.6 Mt of lower grade mineralization had been stockpiled. A total of approximately 230.4 Mt had been excavated including 18.4 Mt of overburden and 134.7 Mt of waste rock.

1.5 Geology and Mineralization

The Troilus Gold deposit lies within the eastern segment of the Frotêt-Evans Greenstone Belt (FEGB), in the Opatica Subprovince of the Superior Province in Québec. The FEGB is largely dominated by tholeiitic basalts and magnesian basalts that occur in association with felsic and intermediate calc-alkaline pyroclastic rocks, lava flows, and local ultramafic layers. Syn- to post-deformational gabbroic to monzogranitic plutonic rocks occur throughout the greenstone belt.

The main mineralized zones at the Property occur around the margins of the Troilus Diorite, and comprise the Z87 Zone, J Zone, X22 Zone, and the SW Zone. Other significant targets near these zones include: the northern continuity of the J Zone, the Allongé Zone; the northeastern continuity of the SW Zone, the Gap Zone.

Troilus is primarily an Au-Cu deposit, but contains minor amounts of Ag, Zn and Pb, as well as traces of Bi, Te, and Mo. Gold-copper mineralization at the Troilus deposit comprises two distinct styles, disseminated and vein-hosted. Gold mineralization is spatially correlated with the presence of sulphides, even though the sulphide content does not directly correlate with gold and copper grade. The matrix of the diorite breccia, the diorite and the felsic dikes represent the main host rocks for the mineralized intervals.

1.6 Deposit

The mineralized zones of the Project are known as Archean porphyry-type deposits. Other interpretations of the deposits include superimposed, structurally controlled, “orogenic” gold deposits.

1.7 Exploration

Since the formation of Troilus, exploration activities have been focussed on developing the principal mineralized zones: Z87, J, X22 and SW Zones. Troilus has also been active at various exploration targets along strike of these zones to the northeast (Allongé, Carcajou), between (Gap Zone), and the southwest (Beyan, Cressida). Regional exploration targets include the Testard Target and the Pallador and Rocket Targets.

In 2018 and 2019, field mapping and prospecting work supported Troilus’ team to improve the understanding of the lithological and structural controls on gold mineralization across the property and confirmed the overall potential for extending the current known limits of the main mineralized zones. In 2018, Troilus retained SRK Consulting (Canada) Inc. (SRK) to conduct a structural geology investigation at the Project. The study focused on the exposed geology in the Z87 Zone open pit and the J Zone open pit.

In June 2020, Troilus completed a preliminary field exploration program applying the regional structural and geological models to areas along strike, and south, of the known deposits. In 2020 and 2021, Troilus completed two high-resolution magnetic geophysical surveys were completed. Initial results have led to the discovery several areas of interest that have been actively explored between 2020 and 2023. These targets include: the Beyan target and Cressida target, situated approximately 8 km and 14 km, respectively, southwest and along strike of the SW Zone of the Project; The Testard and Freegold-Bullseye target situated approximately 10 km south of the SW Zone; and the Pallador target situated approximately 35 km south of the Troilus mine.

Each of these target areas have been subject to, in varying degrees, geological mapping and prospecting programs, ground geophysical surveys and exploration drill hole programs.

1.8 Drilling

Since 1986, there have been several drilling programs completed on the Property by previous owners. There was no drilling on the Property from 2008 to 2017 and Troilus’ drill programs were completed from 2018 to 2023. Most of the 2018 and 2019 drill holes targeted the Z87 and J Zones at depth and along strike. Initial drilling in 2019 led to the discovery off the SW Zone that has largely been the focus of drill programs in 2021 and 2022.

The resource drill hole database contains 1,492 surface drill holes totalling approximately 449,168 m mainly in the Z87, J, X22 and SW Zones, and includes exploration and geotechnical drill holes.

AGP considers the drilling was undertaken in accordance with industry standards and best practices without any major adverse aspects that could have materially impacted the accuracy and reliability of the mineral resource estimate.

1.9 Sample Preparation, Analyses and Security

AGP reviewed the QA/QC program and is of the opinion it is in accordance with standard industry practice and CIM Exploration Best Practice Guidelines. Troilus personnel have taken all reasonable measures to ensure the sample analysis completed is accurate and precise. AGP considers the assay results and database acceptable for use in the estimation of mineral resources.

AGP is of the opinion that the preparation and analyses are satisfactory for this type of the deposit and that the sample handling and chain of custody meet or exceed industry standards.

1.10 Data Verification

AGP received the database containing all drill holes for the Z87 Zone, J Zone, X22 Zone, and SW Zone in a Leapfrog project that included, but not limited to, collar, survey, assay, and lithology files.

An export of the Geotic database was received for data validation and QA/QC review. AGP verified approximately 7.5% of the data from the 2021 and 2022 drill programs (approximately 13,000 records out of 175,000) and included data across all three zones. The gold, copper, silver assay values, and density values, were compared to the laboratory certificates provided to Troilus by ALS. No errors were found.

AGP is of the opinion the database is representative and adequate to support the mineral resource estimates for the Troilus deposits. AGP is satisfied that the core descriptions, sampling procedures, and data entries were conducted in accordance with industry standards.

1.11 Mineral Processing and Metallurgical Testing

Recent samples from the J, Southwest (SW), 87 and X22 pits were submitted to various testing facilities for metallurgical testing in support of the current studies. The findings are as follows:

- Hardness testwork results classified Troilus ore to be competent with A x b value of 26.0 at the 15^{th} percent and 29.8 at the 50th percentile.
- Bond abrasion index measuring from 0.2 to 0.4 indicates that the ore is moderately abrasive.
- Crushing work index has been derived from A x b data to be 22.5 kWh/t.
- Bond ball mill work index of 13.8 kWh/t at the 85^{th} percentile and 12.1 kWh/t at the 50th percentile.
- Locked cycle PILOTWAL HPGR testwork resulted in an average m dot value of 270 $t{\cdot}s/m^3{\cdot}h$ at a net pressing force of 3.33 N/mm^2.
- The gravity gold recovery (future) is expected to be ~32%, similar to the gravity recovery achieved in the historical Troilus operation.
- Optimum flotation grind size was P80 75µm for primary milling and P80 20µm for regrind milling.
- Further treatment (leaching) of the flotation tails is not required or justifiable economically due to low flotation tails grades.

- Based on LOM head grade of 0.49 g Au/t, 1.00 g Ag/t and 0.06% Cu, the following recoveries are expected:
  - 91.6% Cu, 91.9% Au and 86.6% Ag for J-Zone
  - 89.9% Cu, 87.4% Au and 82.7% Ag for SW-Zone
  - 91.8% Cu, 94.7% Au and 97.6% Ag for Zone 87
  - 94.5% Cu, 93.1% Au and 89.9% Ag for X22 Zone
- Flotation reagent consumptions for all zones combined are approximately 56 g/t KAX, 32 g/t SPRI 206, 29 g/t frother and between 100 to 400 g/t Na_2SO_3 depressant.

1.12 Mineral Resources

The mineral resources for the Project include the four principal mineralized zones: Z87, J, X22 and SW Zones. The mineral resource estimates were prepared and disclosed in accordance with the CIM Definitions for Mineral Resources and Mineral Reserves (2014). The Qualified Person (QP) responsible for the mineral resource estimates is Mr. Paul Daigle, P.Geo., géo., Principal Resource Geologist for AGP. The effective date of these mineral resource estimates is 2 October 2023.

The mineral resource estimates were prepared using interpreted mineralized domains based on a gold equivalent (AuEQ) of greater than 0.3 g/t AuEQ at each of the four zones. The block models for each deposit all use a block model matrix of 5 m x 5 m x 5 m and gold, copper and silver grades were estimated using ordinary kriging interpolation method on capped composite values. Table 1-1 and Table 1-2 present a summary of the mineral resource estimates for the Project.

Table 1-1: Open Pit Mineral Resources for the Troilus Project at a 0.3 g/t AuEQ Cut-off Grade – All Zones

		Grade				Contained Metal			
Class	Tonnes (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	AuEQ (g/t)	Au (Moz)	Cu (Mlb)	Ag (g/t)	AuEQ (Moz)
Z87									
Indicated	197.1	0.67	0.07	1.21	0.80	4.21	320.69	7.67	5.04
Inferred	37.1	0.59	0.06	1.11	0.70	0.71	50.17	1.33	0.84
JZ									
Indicated	151.9	0.50	0.06	0.96	0.61	2.45	215.71	4.71	2.98
Inferred	24.2	0.46	0.07	0.94	0.57	0.35	35.37	0.73	0.44
X22									
Indicated	59.2	0.51	0.06	1.24	0.62	0.98	79.34	2.35	0.19
Inferred	13.6	0.53	0.07	1.48	0.67	0.23	21.76	0.65	0.29
SW									
Indicated	98.0	0.50	0.05	0.94	0.60	1.59	109.91	2.94	1.89
Inferred	1.6	0.37	0.04	0.96	0.45	0.02	1.36	0.05	0.02
TOTALS – ALL ZONES									
Indicated	506.2	0.57	0.07	1.09	0.68	9.23	725.66	17.67	11.11
Inferred	76.5	0.53	0.06	1.12	0.65	1.31	108.66	2.75	1.59

Notes:

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Summation errors may occur due to rounding.

Open pit mineral resources are reported within optimized constraining shells.

Open pit cut-off grade is 0.3 g/t AuEQ.

AuEQ equivalents were calculated as follows:

Z87 Zone AuEQ = Au grade + 1.5628 * Cu grade + 0.0128 * Ag grade
J4/J5 Zone AuEQ = Au grade + 1.5107 * Cu grade + 0.0119 * Ag grade
SW Zone AuEQ = Au grade + 1.6823 * Cu grade + 0.0124 * Ag grade
X22 Zone AuEQ = Au grade + 1.5628 * Cu grade + 0.0128* Ag grade

Metal prices for the AuEQ formulas are: US$ 1,850/ oz Au; $4.25/lb Cu, and $25.00/ oz Ag; with an exchange rate of US$1.00:CAD$1.30

Metal recoveries for the AuEQ formulas are:

Z87 Zone 95.5% for Au recovery, 94.7% for Cu recovery and 98.2% for Ag recovery
J Zone 93.1% for Au recovery, 89.3% for Cu recovery and 88.9% for Ag recovery
SW Zone 85.7% for Au recovery, 91.5% for Cu recovery and 85.6% for Ag recovery
X22 Zone 95.5% for Au recovery, 94.7% for Cu recovery and 98.2% for Ag recovery

Capping of grades varied between 2.30 g/t Au and 21.00 g/t Au; between 0.06% cu and 4.36 %Cu, and between 3.20 g/t Ag and 55.00 g/t Ag; on raw assays.

The density (excluding overburden and fill) varies between 2.64 g/cm^3 and 2.93 g/cm^3 depending on lithology for each zone.

Table 1-2: Underground Mineral Resources for the Troilus Project at a 0.9 g/t AuEQ Cut-off Grade – All Zones

		Grade				Contained Metal			
Class	Tonnes (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	AuEQ (g/t)	Au (Moz)	Cu (Mlb)	Ag (g/t)	AuEQ (Moz)
Z87									
Indicated	0.5	1.59	0.15	0.54	1.83	0.02	1.55	0.01	0.03
Inferred	1.1	1.99	0.12	0.46	2.19	0.07	2.96	0.02	0.08
J Zone									
Indicated	0.2	1.21	0.07	1.46	1.33	0.01	0.29	0.01	0.01
Inferred	1.0	1.25	0.05	0.99	1.34	0.04	1.13	0.03	0.04
X22									
-none-									
-none-									
SW									
Indicated	1.4	1.28	0.07	2.44	1.42	0.06	2.00	0.11	0.06
Inferred	1.9	1.05	0.06	16.62	1.37	0.06	2.66	1.01	0.08
TOTALS – ALL ZONES									
Indicated	2.1	1.35	0.09	1.90	1.51	0.09	3.84	0.13	0.10
Inferred	4.0	1.36	0.08	8.21	1.58	0.18	6.75	1.06	0.20

Notes:
Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
Summation errors may occur due to rounding.
Underground resources reported in 0.9 g/t AuEQ grade shells
Underground cut-off grade is 0.9 g/t AuEQ.
AuEQ equivalents were calculated as follows:

Z87 Zone	AuEQ = Au grade + 1.5628 * Cu grade + 0.0128 * Ag grade
J4/J5 Zone	AuEQ = Au grade + 1.5107 * Cu grade + 0.0119 * Ag grade
SW Zone	AuEQ = Au grade + 1.6823 * Cu grade + 0.0124 * Ag grade
X22 Zone	AuEQ = Au grade + 1.5628 * Cu grade + 0.0128* Ag grade

Metal prices for the AuEQ formulas are: US$ 1,850/ oz Au; $4.25/lb Cu, and $25.00/ oz Ag; with an exchange rate of US$1.00: CAD$1.30.
Metal recoveries for the AuEQ formulas are:

Z87 Zone	95.5% for Au recovery, 94.7% for Cu recovery and 98.2% for Ag recovery
J Zone	93.1% for Au recovery, 89.3% for Cu recovery and 88.9% for Ag recovery
SW Zone	85.7% for Au recovery, 91.5% for Cu recovery and 85.6% for Ag recovery
X22 Zone	95.5% for Au recovery, 94.7% for Cu recovery and 98.2% for Ag recovery

Capping of grades varied between 2.30 g/t Au and 21.00 g/t Au; between 0.06% cu and 4.36 %Cu, and between 3.20 g/t Ag and 55.00 g/t Ag; on raw assays.
The density (excluding overburden and fill) varies between 2.64 g/cm^3 and 2.93 g/cm^3 depending on lithology for each zone.

AGP is not aware of any information not already discussed in this report, which would affect their interpretation or conclusions regarding the subject property. AGP is required to inform the public that the quantity and grade of reported Inferred mineral resources in this estimation must be regarded as conceptual in nature and are based on limited geological evidence and sampling. The geological evidence is sufficient to imply, but not verify, geological grade or quality of continuity. For these reasons, an Inferred mineral resource has a lower level of confidence than an Indicated mineral resource. It is reasonably expected that most of the Inferred Mineral Resources could be upgraded to

Indicated Mineral Resources with continued exploration. The rounding of values, as required by the reporting guidelines, may result in apparent differences between tonnes, grade, and metal content.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

1.13 Mineral Reserves Estimates

The Project is planned to be an open pit operation using conventional mining equipment. All work is based on the mine plans generated by AGP.

Costs are based on first principles build-up of operating and capital costs for the life of the project with current vendor quotations for consumables and maintenance. Mining capital costs were based on vendor submissions.

The mineral reserves for the Project are based on the conversion of the Measured and Indicated mineral resources in the current mine plan within the J, 87, X22, and Southwest (SW) open pits. No measured mineral resources are contained in these three resource models, so there will be no proven reserves. Indicated mineral resources are converted directly to probable reserves. The estimates were prepared under the supervision of Willie Hamilton, P.Eng. of AGP, a QP as defined under NI 43-101.

The mineral resources in the underground areas below the pits are also not considered in the mineral reserves at this time.

The total mineral reserves for the Project are shown in metric units in Table 1-3.

Table 1-3: Proven and Probable Reserves – January 15,2024

Reserve Class	Tonnage (Mt)	Grades Au (g/t)	Cu (%)	Ag (g/t)	AuEq (g/t)	CuEq (%)	Contained Metal Au (Moz)	Cu (Mlb)	Ag (Moz)	AuEq (Moz)	CuEq (Blbs)
Proven	0.0	0.00	0.000	0.00	0.00	0.00	0.00	0	0.00	0.00	0.00
Probable	380	0.49	0.058	1.00	0.59	0.39	6.02	484	12.15	7.26	3.24
Proven and Probable	**380**	**0.49**	**0.058**	**1.00**	**0.59**	**0.39**	**6.02**	**484**	**12.15**	**7.26**	**3.24**

Note: This mineral reserve estimate has an effective date of January 15, 2024, and is based on the mineral resource estimate dated October 2, 2023, for Troilus by AGP Mining Consultants Inc. The mineral reserve estimate was completed under the supervision of Willie Hamilton, P.Eng. of AGP, who is a Qualified Person as defined under NI 43-101. Mineral Reserves are stated within the final pit designs based on a US$1,550/oz gold price, US$20.00/oz silver price and US$3.50/lb copper price. An NSR cut-off of C$9.96/t was used to define reserves. The life-of-mine mining cost averaged C$3.99/t mined, preliminary processing costs were C$8.02/t ore and G&A was C$1.94/t ore placed. The metallurgical recoveries were varied according to gold head grade and concentrate grades. 87 pit recoveries for equivalent grades were 95.5%, 94.7% and 98.2% for gold, copper, and silver, respectively. J pit recoveries for equivalent grades were 93.1%, 89.3% and 88.9% for gold, copper, and silver, respectively. X22 pit recoveries for equivalent grades were 95.5%, 94.7% and 98.2% for gold, copper, and silver, respectively. SW pit recoveries for equivalent grades were 85.7%, 91.5% and 85.6% for gold, copper, and silver, respectively. The formulas used to calculate equivalent values are as follows, for 87 Pit AuEq = Au + 1.5361*Cu +0.0133 *Ag, for J Pit AuEq = Au + 1.4849*Cu +0.0123 *Ag, for SW Pit AuEq = Au + 1.6535*Cu +0.0129 *Ag, for X22 Pit AuEq = Au + 1.5361*Cu +0.0133 *Ag.

The QP has not identified any known legal, political, environmental, or other risks that would materially affect the potential development of the Mineral Reserves.

Risks that could materially affect the mineral reserve estimate include mining selectivity near the ore contacts, slope stability and assumed process recoveries for given rock types. These are considered manageable risks which will be mitigated as more testwork, and operating experience is obtained.

1.14 Mining Methods

1.14.1 Mine Geotechnical and Dewatering

The stability analyses indicate that the rock mass is favourable to the development of steep inter-ramp slopes in all areas of the pits except for open pit east walls where the bench face angle is constrained by planar failure involving moderately dipping foliation. Recommended open pit east wall inter-ramp angles range from 45 to 50 degrees, except at X22 pit, where the foliation is shallower, and the recommended inter-ramp angle is 38 degrees. For other wall orientation, the recommended inter-ramp angles range from 50 to 57 degrees.

Building on the precedents and experience gained by Inmet operation while mining the historical 87 and J4 pits, the pit slope designs were optimized assuming that ground support will be used to achieve safe bench geometries. The implementation of the slope designs will require a dedicated team, strong procedures and systems and capability to adapt. It will require a high level of skill applied to perimeter blasting, scaling ground support, depressurization, and slop movement monitoring by the mine team.

Pit dewatering is an important part of mining at Troilus particularly since the pit will be below the creek level and is currently full of water. Efficient and cost-effective dewatering will play a role in the Project development. Dewatered slopes may allow a reduction in the strip ratio by permitting steeper inter-ramp angles that would also be inherently safer.

It is estimated that an average of 8.4 Mm^3/year of runoff (4.3 Mm^3/year) and seepage (4.1 Mm^3/year) will need to be pumped from within the pits (WSP, 2024c). From there, it will need to be pumped to the required discharge point near the settling ponds. Storm events have the potential to impact mining operations, and between 0.3 m^3/s and 0.5 m^3/s per pit of pumping capacity would be required for a brief period of time to recover from one of these storm events (WSP, 2024c).

1.14.2 Open Pit Mining

Pit designs were developed for the J, 87, X22, and SW pit areas. The J pit design consists of two phases of successive pushbacks around the entire pit perimeter. The 87-pit design includes an initial phase 0 at the south end to assist with site water management, followed by phases 1 to 3 in the main portion of the pit. The X22 pit design consists of two phases, with slightly higher grades in the phase 1 at the south. The SW pit design consists of two phases which can be scheduled as satellite phases from the northern pits. The pit optimization shells used to guide the ultimate pits were also used to outline areas of higher value for targeted early mining and phase development. All pits were developed using 10 metre bench heights.

Tonnes and grade for the designed pit phases are reported in Table 1-4 using the diluted tonnes and grade from the models and a mining recovery of 98% to account for additional mill feed losses.

Table 1-4: Pit Phase Tonnages and Grades

Pit	Phase	Ore (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	NSR (C$/t)	Waste (Mt)	Total (Mt)	Strip Ratio
J	1	74.4	0.45	0.06	0.91	29.53	153.0	227.4	2.1
	2	50.8	0.42	0.058	0.84	27.79	164.7	215.5	3.2
J Total		**125.2**	**0.44**	**0.058**	**0.88**	**28.82**	**317.7**	**442.9**	**2.5**
87	0	1.6	0.65	0.04	0.95	42.20	8.5	10.1	5.3
	1	31.6	0.55	0.062	1.17	37.09	139.3	170.9	4.4
	2	69.0	0.58	0.068	1.14	39.38	179.5	248.5	2.6
	3	63.9	0.52	0.055	1.08	34.26	272.0	335.9	4.3
87 Total		**166.1**	**0.55**	**0.062**	**1.12**	**37.00**	**599.4**	**765.5**	**3.6**
X22	1	16.5	0.43	0.07	1.61	29.59	56.5	73.0	3.4
	2	20.0	0.40	0.047	0.79	25.48	53.1	73.0	2.7
X22 Total		**36.4**	**0.41**	**0.058**	**1.16**	**27.34**	**109.6**	**146.0**	**3.0**
SW	1	34.0	0.48	0.05	0.75	29.09	75.1	109.0	2.2
	2	17.9	0.52	0.035	0.78	30.67	69.2	87.1	3.9
SW Total		**51.9**	**0.49**	**0.045**	**0.76**	**29.64**	**144.3**	**196.1**	**2.8**
Troilus Total		**380**	**0.49**	**0.058**	**1.00**	**32.37**	**1,171**	**1,550**	**3.1**

The mine schedule for open pit mining consists of 380 Mt of mill feed grading 0.49 g/t gold, 0.058% copper, and 1.0 g/t silver providing mill feed for 22 production years. Open pit waste tonnage totals 1,171 Mt and will be placed into waste storage areas. The overall open pit strip ratio is 3.1:1. The mine schedule utilizes the pit phases described previously to send a maximum of 18.3 Mtpa (50,000 tonnes per day) of feed to the mill facility.

The current mine life includes two years of pre-stripping followed by twenty-one years of mining. A maximum descent rate of 9 benches per year per phase was applied for open pit mining to ensure that reasonable mining operations and mill feed control would occur. Peak primary mining rates are scheduled at approximately 87 Mt per year between years 5 to 8. Mill feed is stockpiled during the pre-production years, with approximately 0.7 Mt of feed for plant commissioning. A peak stockpile capacity of 48 Mt was reached near the end of year 11. Stockpile material is reclaimed from stockpiles after completion of mining and continues until early into the 22nd year.

1.15 Recovery Methods

The plant has been designed for a nominal throughput of 50,000 t/d (dry) with the following process flowsheet:

- two-stage crushing using an open circuit gyratory crusher followed by two parallel closed-circuit secondary cone crushers to produce a -45 mm crushed product for feed to the HPGR; the crushed ore will be stored in a covered stockpile
- a single HPGR operating in closed circuit with four parallel vibrating screens to produce a -5 mm feed to the ball mills

- grinding and classification in two parallel closed ball milling circuits with provision for rougher & scavenger gravity concentration in the future
- bulk rougher and scavenger flotation to produce a primary copper concentrate
- bulk concentrate regrinding in an open-circuit configuration, with provision for regrind gravity concentration in the future
- bulk cleaner flotation, using three stages of column cleaning
- final copper concentrate thickening and filtration
- smelting of rougher scavenger and cleaner circuit gravity concentrates to produce doré in the future
- tailings thickening of the combined flotation tails and disposal in a tailings storage facility (TSF)

1.16 Project Infrastructure

The Project includes various plant processing facilities, mining facilities, site-wide infrastructure and on-site facilities that will be newly constructed, as listed in Section 18.1. In addition to the newly constructed facilities, there is existing infrastructure that will be repurposed for the Project, as specified in Table 18-1.

A water management plan, including a feasibility engineering design and site-wide water balance model, was completed in view of restarting mining operations on the Troilus Site. (the Site). The water management plan aimed to facilitate efficient mine operations and reduce effects on downstream receiving waterbodies. The proposed water management plan includes water management structures to construct over the life-of-mine (LoM).

The tailings storage facility will be developed from the existing one which will limit the overall footprint disturbance. This structure will have the capacity to accommodate the first 10.5-year life of mine production and then from years 11-22, the tailings will be disposed subsequently into the mined-out SW pit, J pit and 87 pit as described in this FS. Waste rock from the mine operation placed along the tailings storage facility's containment dyke will enhance the facility's stability and safety and will also limit the footprint disturbance.

1.17 Market Studies and Contracts

This Report uses the following metal prices for the base case economic analysis:

- Gold $1,975 US/oz
- Copper $4.05 US/lb
- Silver $23.00 US/oz

Based on past operating experience, the mine's concentrate will be a clean product that will be in demand for its contained gold. While the average copper content is lower than standard concentrates with its 13.3% grade, the higher gold grade of 80 g/t will make it attractive to various worldwide smelters and local smelters.

Indicative smelter terms were provided to Troilus for the FS that have been incorporated in the study. No definitive smelter agreements have been obtained for the concentrate, although, the concentrate would not be difficult to market according to the commissioned study. This is due in part to the higher gold grade in the copper concentrate and apparent lack of deleterious elements. No penalties need to be applied in the terms for the concentrate.

1.18 Environmental, Permitting, and Social Considerations

The Site was previously exploited from 1998 to 2011 and was partially rehabilitated from 2011 until now. This gives the advantage of having a lot of real data from which to assess the impacts and effects of future exploitation with precision.

The Site has currently two environmental statuses: exploration and closed (reclaimed) site. The site has been reclaimed starting at the end of the previous operation. Buildings and infrastructures have been dismantled. Soils have been characterized and reclaimed. The WMF and the TSF have been revegetated.

In August 2020, Troilus received a Certificate of Authorization from MELCC to proceed with dewatering of the J4 and 87 pits and will begin dewatering in 2024.

Troilus keeps good relations and has frequent exchange sessions with the Cree Nation of the Eeyou-Istchee James Bay Region, and in particular the Cree Nation of Mistissini, the First Nations community whose traditional land use and economic activities may be most directly impacted by the Troilus' development. Troilus maintains a community liaison office in, communicates regularly with impacted families, the Chief and Council in Mistissini and other community organizations such as the Cree Mineral Board, the Cree Trappers Association, and the Board of Education to keep the community apprised of developments.

The opening and operation of a mine triggers the environmental impact assessment and review procedure under chapter II of the Environment Quality Act (EQA). This chapter covers the particular regime defined by the James Bay and Northern Québec Agreement (JBNQA). The process includes a participation by the Natives so that they can protect the rights and guarantees granted to them under the Agreement. On the Federal side, planning to open a mine with an ore production capacity more than 5,000 t/day subjects the Project to the IAA.

The two process have started with the filing of The Project notice and the Detailed Project Description in May 2022. Guidelines for both processes have been received and writing of the ESIA is ongoing.

1.19 Capital and Operating Costs

1.19.1 Capital Costs

The capital cost estimate (estimate) includes all the direct and indirect costs along with the appropriate estimating contingencies for all the facilities required to bring the Project into production, as defined by this Report. All equipment and material are assumed to be new. Labour costs based on the statutory laws governing benefits to workers in effect in Québec at the time of the estimate. The estimate does not include any allowances for scope changes, escalation, and exchange rate fluctuations. The execution strategy is based on an engineering, procurement, and construction management (EPCM) implementation approach with Troilus self performing the construction management and horizontal (discipline based) construction contract packaging.

The capital cost estimate for the Project was developed to provide an estimate suitable for a Feasibility Study including cost to design, construct, and commission the facilities. The estimate produced is described as a Class 3 with an expected accuracy of +15% -10%. This classification is based on the AACE international standard.

The total capital costs for the Project are estimated to be $1,074.6 million dollars expressed in H2 2023 price levels exclusive of duties and taxes as shown in Table 1-5. The mining costs including financing of the mine fleet which reduces the initial capital cost and transfers that cost to operating.

Table 1-5: Troilus Gold Copper Project Capital Cost Estimate

Area	Initial Capital (M$)	Sustaining Capital (M$)	Total Capital (M$)
Open Pit – Prestrip (capitalized)	213.0	-	213.0
Open Pit - Capital	45.3	99.3	144.6
Open Pit Mining - Subtotal	258.3	99.3	357.6
Processing	443.0	15.1	458.1
Infrastructure	100.3	27.7	128.0
Environmental	10.7	67.4	78.2
Indirects	173.0	50.5	223.4
Contingency	89.3	16.6	105.9
Total	**1,074.6**	**276.6**	**1,351.2**

1.19.2 Operating Costs

The estimated life of mine operating costs are shown in Table 1-6 below.

Table 1-6: Troilus Gold Copper Project Operating Costs (Year 1 – 21)

Cost Area	Cost (M$)	Unit Cost ($/t Mill Feed)
Open Pit Mining	4,394.5	11.60
Processing	2,135.4	5.64
General and Administration	430.5	1.14
Concentrate Trucking	119.4	0.32
Port Costs	32.9	0.09
Shipping to Smelter	107.1	0.28
Total Operating Cost	**7,079.8**	**19.06**

Diesel and electricity pricing was obtained locally and are $1.07/l and $ 26/MWh, respectively. The mine equipment is a mix of diesel (trucks and loaders) and electrical (shovels and drills) powered equipment.

The mining cost includes the financing cost of $ 0.51/t moved life of mine or $ 1.92/t milled.

General and Administrative costs consider a camp operation with the cost of leasing the permanent camp in the overall cost.

1.20 Economic Analysis

A pre-tax and post-tax cash flow model has been prepared by AGP on behalf of Troilus for the evaluation of the Project.

All pricing is H2 2023 United States Dollars (USD) unless otherwise noted.

The results indicate a post-tax NPV (5%) of $885 million with an IRR of 14.0% and a payback period of 5.7 years. Initial capital is $1,074.6 million with life of mine capital totaling $1,351.2 million.

The results of the financial analysis using a discounted cash flow are summarized in Table 1-7.

Table 1-7: Troilus Gold Copper Project – Discounted Cash Flow Financial Summary

Parameter	Units	Pre-Tax	Post-Tax
Metal Prices			
Gold	US$/oz	1,975	
Copper	US$/lb	4.05	
Silver	US$/oz	23	
Exchange Rate	C$:US$	0.74	
Net Present Value (5%)	US$ M	1,564	884.5
Internal Rate of Return	%	18.1	14.0
Net Revenue less Royalties	US$ M	12,122.0	12,122.0
Total Operating Cost	US$ M	7,224.2[1]	7,224.2[1]
Life of Mine Capital Cost	US$ M	1,351.2	1,351.2
Taxes	US$ M	-	1,342.0
Net Cash Flow	US$ M	3,546.5	2,204.6
Payback Period	Years	5.4	5.7
Cash Costs (with credits)	US$/oz	1,064	1,313
All-in Sustaining Cost	US$/oz	1,109	
Payable Metals (Life of Mine)			
Gold	Moz	5.38	
Copper	M Lbs	381.8	
Silver	Moz	9.45	
Initial Capital	US$ M	1,074.6	
Sustaining Capital	US$ M	276.6	
Total Capital	US$ M	1,351.2	
Mine Life	Years	21	

[1] Includes the processing cost in Year -1 which is not capitalized.

1.21 Conclusions

The Project is made up of four principal mineralized zones: Z87 Zone, J Zone, X22 Zone, and SW Zone. The Z87 Zone and J Zone were subject to open pit mining operations between 1996 to 2010. It has been established that there are still significant open pit and underground mineral resources in these, and adjacent zones. The X22 Zone has been recently discovered and developed in 2023 and is situated adjacent to the southwest of Z87 Zone. The SW Zone, situated approximately 2.5 km southwest of the Z87 Zone, has been the focus of several drill campaigns since 2019 and has been established as a significant deposit for the Project. The gold grades within the interpreted mineralized domains are continuous and may still be open along strike and at depth.

The mineralized zones on the Property occur around the margins of the Troilus Diorite and comprise the Z87 Zone, J Zone, and X22 Zone. The SW Zone lies along strike and southwest of the Z87 Zone. Other important mineralization discovered on the Property to date include: the northern continuity of the J Zone, in the Allongé Target and Carcajou Target; and the north-western continuity of the SW Zone, toward Z87 Zone, the Gap Zone; and to the southwest of the SW Zone, in the Beyan and Cressida Targets. Additionally, Troilus has also investigated several regional exploration targets on the Property that include: the Testard Target, the Freegold-Bullseye Target, and the Pallador Target.

The Project is primarily a gold-copper deposit, but contains minor amounts of Ag, Zn and Pb, as well as traces of Bi, Te, and Mo. The gold and copper mineralization at the Troilus deposit comprises two distinct styles, disseminated and vein hosted. Gold mineralization is spatially correlated with the presence of sulphides, even though the sulphide content does not directly correlate with gold and copper grade. The matrix of the diorite breccia, the diorite and the felsic dikes represent the main host rocks for the mineralized intervals.

AGP concludes that further development of the mineralized zones is warranted and recommended.

1.22 Recommendations

1.22.1 Summary

The QPs recommend that Troilus Gold proceed with advancing the Troilus Gold and Copper Project to Basic Engineering as part of the Project development plan. Recommendations and associated budgets are provided by the QPs to carry this work forward.

Estimated costs by area are provided in Table 1-8.

Table 1-8: Recommended Budget for Basic Engineering

Area of Study	Approximate Cost (C$)
Geology	$2,200,00
Geotechnical	$357,000
Mining	$150,000
Mineral Processing and Metallurgy	$25,000
Infrastructure (long lead items, engineering)	$63,700,000
Environmental	$2,750,000
TOTAL	**$69,182,000**

1.22.2 Geology

AGP recommends:

- delineation drilling continue on all four mineralized zones of the Project to define the limits of each zone along strike (approximately 6,000 m)
- that the twinning of historic, pre-2018, drill holes, be targeted with more current drill information (approximately 3,000 m)
- that bulk density and assay analysis for silver be completed for the initial drilling at Z87 Zone (approximately 4,000 samples)

The estimated budget for this development work is estimated to be C$2.2 million.

1.22.3 Geotechnical

WSP recommends:

- continue collecting oriented structural data in exploration holes
- plan two 300 m deep geotechnical holes in areas where uncertainty in exploration data orientation could have impacts on the slope design, such as the southeastern and northeastern portion of J4 pit
- plan two 200 m long geotechnical boreholes to improve characterization at the SW pit and to distinguish weaker conditions observed in the felsic and breccia units from potentially stronger and better quality intermediate volcanics (west wall) and mafic volcanics (east wall)
- review of the data, structural model update, design update, and potential added discretization of the design domains that the new data would allow
- review of the data, structural model update, design update, and potential added discretization of the design domains that the new data would allow

1.22.4 Mining

AGP recommends:

- Blast Optimization - using detailed rock information
- Equipment Selection - review equipment selection and fine tune drill size, shovel buckets and truck boxes to handle the abrasive materials and maximize carrying capacity/reduce unit costs
- Shovel Bucket Grade Control - examination of shovel bucket-based technology to accurately track grade from face to mill

1.22.5 Processing

Additional testwork is recommended to enhance confidence in the regrind circuit design. This should include:

- settling and rheology testing on flotation concentrates
- regrind milling testwork to obtain a signature plot for the flotation concentrates

The cost associated with these additional testwork is estimated to be $25,000.

Other recommendations include consultation with a concentrate marketing specialist to advise on current penalty and payment terms for minor elements.

1.22.6 Infrastructure

The following is recommended for the next phase in relation to the project infrastructure :

Site Infrastructure

During the Basic Engineering Stage, Lycopodium recommends :

- advancing process, mechanical, and electrical engineering to facilitate the procurement of long lead mechanical and electrical equipment packages. Orders are placed to obtain vendor data, requiring an initial payment of USD 21.1 million (CAD 28.5 million) and a first installment of USD 23.1 million (CAD 31.2 million); these costs, totaling USD 44.2 million (CAD 59.7 million), are included in the capital cost estimate to support further design of critical concrete and steelwork areas
- commencement of Basic Engineering to support long lead item procurement – USD 3.0 million (CAD 4.0 million)

Site Wide Water Management

In preparation for the next engineering phase for the water management structures, WSP recommends completing the following studies:

- extend and improve geotechnical characterization along the water management structures footprints
- extend and improve the available geochemical characterization of the various water streams to be managed by the future Troilus mine operations
- further refine hydrology and aquatic baseline studies to confirm and optimize the fish passage and fish habitat approach and assumptions for the Main Diversion Channel and secondary diversions
- develop progressive reclamation and mine closure plans
- depending on the schedule of the next engineering phase, an update of the climate baseline analysis to consider the most recent climate change projections may be recommended

WSP recommends the optimization activities for the next engineering phase:

- update and review of alignments, profiles, typical cross-sections, and footprints for channels, ditches, sumps, and ponds, to improve hydraulic performance and limit construction costs
- explore the opportunities to optimize pumping capabilities

Tailings Management Facility

WSP recommends:

- identify alternative tailings storing capacity option to cover a shortage of approximately 15 months (22 Mt) occurring around year 17 based tailings production, mine plan and the planned tailings storage capacity

- conduct a dam safety review and dam breach analysis for detailed engineering design and revisit the dam consequence classification
- develop a site-specific seismic hazard assessment to establish the earthquake peak ground acceleration and scaled ground motion time histories and to revisit the liquefaction assessment, dynamic stress-deformation analyses with these updated parameters
- confirm the geochemistry of tailings and waste rock, design changes could be required if the tailings or waste rock are reclassified as high-risk according to Directive 019 classification

Water Treatment

WSP recommends:

- confirm the water treatment design assumptions during the next stages of design to confirm water treatment requirements remain those of the existing water treatment plant

2 INTRODUCTION

This Report was prepared on behalf of Troilus, by AGP.

Troilus is a Canadian exploration company with its corporate office located in Montreal, Canada. Troilus is focused on the development of the Project located in central Québec, situated approximately 170 km north of Chibougamau. The mineral rights for the Property are divided into two projects: the Project and the Troilus Frotêt Project. The mineral rights to the Property cover a total area of approximately 44,124.88 ha. Of this total area, 7,242 ha of this land package is 50% owned by Troilus and 50% owned by Argonaut Gold through a joint venture agreement, with the remainder of the mineral rights being 100% held by Troilus.

The Troilus Mine was originally an open pit mining operation producing gold, copper, and silver continuously from November 1996 to April 2009. The Troilus Mine produced over 2 million oz of gold and approximately 70,000 t of copper. After the mine ceased production in 2009, the 20,000 t/d mill processed low grade stockpiles until the end of June 2010. Following this, the mill was sold and shipped to Mexico and the main camp facilities were dismantled in late 2010. A significant amount of site infrastructure was left in place after the mine closure and disposition of some of the key assets.

2.1 Issuer and Purpose

This Report was prepared on behalf of Troilus, by AGP.

The purpose of the Report is to present the results of the mineral resources and mineral reserves of the four principal deposits of the Project in support of Troilus' Feasibility Study (FS). This Report was prepared in compliance with the Canadian disclosure NI 43-101 and in accordance with the requirements of Form 43-101 F1.

2.2 Qualified Persons

The list of Qualified Persons responsible for the preparation of this Report and the sections under their responsibility are provided in Table 2-1.

Table 2-1: Qualified Persons

Qualified Person	Position	Responsibilities
Paul Daigle, géo.	Principal Resource Geologist AGP Mining Consultants	Section 1.1 – 1.10, 1.12, 4-12, 14, 23, 25.1, 26.1
Marc Rougier, P.Eng.	Senior Principal Geological Engineer WSP	Section 1.14.1, 2.3.2, 16.2, 16.3.10, 16.3.11, 25.2, 26.2
Ryda Peung, P.Eng.	Principal Process Engineer Lycopodium	Section 1.11, 1.15, 13, 17, 21.2, 25.4, 26.4
Willie Hamilton, P.Eng.	Principal Mining Engineer AGP Mining Consultants	Section 1.13, 1.14.2, 15, 16.1, 16.3.(1-9), 25.3, 26.3
Zunedbhai Shaikh, P.Eng.	Senior Mechanical Engineer Lycopodium	Section 1.15,1.16, 18.1 – 18.9, 18.11 – 18.14, 25.5.1, 26.5.1
Laurent Gareau, P.Eng.	Senior Geotechnical Engineer WSP	Section 1.22.6 (TMF), 18.16.1 – 18.16.5, 18.16.7, 25.5.3, 26.5.3
Vlad Rojanschi, P.Eng.	Senior Water Resources Engineer, WSP	Section 1.22.6 (SWWM), 16.3.12, 18.15, 18.16.6, 25.5.2, 26.5.2
Pierre Primeau, P.Eng.	Senior Process Engineer WSP	Section 1.22.6 (WT), 18.15.7, 18.15.8, 25.5.4, 26.5.4
Ann Lamontagne, Eng, Ph.D.	President Lamont	Section 1.18, 20, 25.6, 26.6
Gordon Zurowski, P.Eng.	Principal Mining Engineer AGP Mining Consultants	Section 1.1, 1.17, 1.19.1, 1.19.2, 1.20 – 1.22, 2 to 3, 18.10, 19, 21.1, 21.1.4, 21.2.(1-2)(4-5), 22, 24, 25.7, 25.8 co-author 21.2.4, 21.2.5
Balvinder Singh, P.Eng.	Senior Director of Projects Lycopodium	21.1.2, 21.1.3, 21.1.5 to 21.1.11

2.3 Site Visits and Scope of Personal Inspection

2.3.1 Geology

Mr. Daigle conducted a site visit on 5 October to 7 October 2022 for two days. The site visit included inspection of the drill core logging, sampling, and core storage facilities. A review was made of the logging and sampling procedures and included a review of selected drill core. Selected drill collars were located and recorded to compare with the surveyed drill hole collar locations. Mr. Daigle made a previous site inspection in February 2020.

Mr. Daigle was accompanied on site by:

- Kyle Frank, géo. - Troilus Exploration Manager
- Nicolas Guest, géo. - Troilus Chief Geologist
- Konstantin De Maack, stagiaire - Troilus Project Geologist
- Nicholas Robert-Potvin - Troilus Project Geotechnician

2.3.2 Geotechnical

Mr. Rougier conducted a site visit to the Property from November 9 to 11, 2020. The project site was inspected for 2 days during the site visit.

While on site, Mr. Rougier reviewed geotechnical and exploration core from each pit area, inspected and carried out mapping in existing pit areas and inspected ongoing geotechnical core logging and wireline packer testing. Mr. Rougier was accompanied on site by:

- Andrew Verok, Geotechnical Engineer, WSP-Golder
- Daniel Bergeron, Vice-President, Québec Operations for Troilus Gold Corp.

While on site, Mr. Rougier discussed the structural geology and geological model with

- Bertrand Brassard Chief Geologist for Troilus Gold Corp.

2.3.3 Metallurgy and Processing

Ms. Ryda Peung did not complete a site visit.

2.3.4 Mining

Mr. Willie Hamilton did not complete a site visit.

2.3.5 Infrastructure

Mr. Zunedbhai Shaikh conducted a site visit on 5 and 7 September 2023 for three days. The primary objectives of the site visit were to assess the following existing infrastructure:

1) J4 Pit – Barge and Pumps
2) J4 Pit - Barge and Pumps E-house
3) Lake-A Raw Water Pipeline
4) Modular Water Treatment Plant
5) Permanent Camp – Buried Services
6) HV Substation
7) 25 kVA site Powerlines

2.3.6 Tailings

Mr. Laurent Gareau conducted a site visit between 6 and 7 September 2023 to conduct dam safety inspection of the tailings storage facility.

2.3.7 Water Management

Mr. Vlad Rojanschi visited the Project site between 6 and 7 September 2023. The site visit included an overall site review, including of the existing tailings storage facility and the associated water management structures (water pond, emergency spillway, toe ditches, and effluent points), and the main natural water courses and water bodies (Lake Amont, Lake A, Bibou Creek, and the existing creek diversion channel).

2.3.8 Water Treatment

Mr. Pierre Primeau did not complete a site visit.

2.3.9 Environmental

Ms. Ann Lamontagne conducted a site visit on 5 October to 7 October 2022 for two days. The site visit included an overall site review, the core shack installation, the sites where geochemical research studies have taken place, the existing tailings storage facility and the associated water management structures (water pond and effluent points), and the main natural water courses and some water bodies.

2.3.10 Financial/Mining Costs

Mr. Gordon Zurowski conducted a site visit on 13 July to 15 July 2020 for two days. While on site, Mr. Zurowski reviewed drill core from each pit area, existing pit areas, current infrastructure (tailings, camp, water pipeline, landfill, power line, access roads, diversion ditch) and surrounding geologic prospects.

Mr. Zurowski was accompanied on site by:

- Bertrand Brassard, géo. – former Troilus Chief Geologist
- Daniel Bergeron - Troilus VP Québec Operations
- Jacqueline Leroux - Troilus VP Environment and Permitting

2.3.11 Summary of Site Visits

A summary of the site visits is shown in Table 2-2.

Table 2-2: Dates of Site Visits

Name	Site Visit	Dates
Paul Daigle	Yes	October 5-7, 2022
Marc Rougier	Yes	November 9 – 11, 2020
Ryda Peung	No	N/A
Willie Hamilton	No	N/A
Zunedbhai Shaikh	Yes	September 5-7, 2022
Laurent Gareau	Yes	September 6-7, 2023
Vlad Rojanschi	Yes	September 6-7, 2023
Pierre Primeau	No	N/A
Ann Lamontagne	Yes	October 5-7, 2022
Gordon Zurowski	Yes	July 13 – 15, 2020
Balvinder Singh	No	N/A

2.4 Effective Dates

The final drill hole database was received on 31 August 2023. The effective date of the Mineral Resources for this report is 2 October 2023. The effective date of the Mineral Reserves is January 15, 2024.

2.5 Previous Technical Reports

The Troilus Mine and Troilus Project has been the subject of several technical reports. The previous NI 43-101 technical reports filed on SEDAR are summarized in Table 2-3 below:

Table 2-3: Summary of Previous Technical Reports

Reference	Date	Company	Name
Balint et al., 2003	Apr 24, 2003	Inmet Mining Corp.	Technical Report on the Mineral Resource and Mineral Reserve Estimates at the Troilus Mine, Québec
RPA, 2014	Jun 30, 2014	Copper One Inc.	Technical Report on the Troilus Gold-Copper Mine Mineral Resource Estimate, Québec, Canada,
RPA, 2016	Jun 30, 2016	Sulliden Mining Capital Inc.	Technical Report on the Troilus Gold-Copper Mine Mineral Resource Estimate, Québec, Canada,
RPA, 2017	Nov 20, 2017	Pitchblack Resources Ltd.	Technical Report on the Troilus Gold-Copper Mine Mineral Resource Estimate, Québec, Canada
RPA, 2019a	Jan 1, 2019	Troilus Gold Corp.	Technical Report on the Troilus Gold-Copper Mine Mineral Resource Estimate, Québec, Canada
RPA, 2019b	Dec 20, 2019	Troilus Gold Corp.	Technical Report on the Troilus Gold-Copper Mine Mineral Resource Estimate, Québec, Canada
AGP, 2020a	Aug. 27, 2020	Troilus Gold Corp.	Technical Report and Mineral Resource Estimate on the Troilus Gold-Copper Project, Québec, Canada
AGP, 2020b	Oct. 14, 2020	Troilus Gold Corp.	Preliminary Economic Assessment of the Troilus Gold Project, Québec, Canada
AGP, 2023	Oct. 2, 2023	Troilus Gold Corp.	Technical Report and Mineral Resource Estimate on the Troilus Gold-Copper Project, Québec, Canada

All other technical reports and information used in this report are listed in Section 27.0 References.

2.6 Units of Measure

All units of measurement in this report are in metric and all costs are expressed in United States dollars (USD) unless otherwise stated. Contained gold and silver is expressed as troy ounces (oz). All material tonnes are expressed as dry tonnes (t) unless stated otherwise.

Table 2-4 shows Units of Measure used in this study

Table 2-4: Units of Measure

Unit	Abbreviation	Unit	Abbreviation
Above mean sea level	amsl	Acre	ac
Ampere	A	Annum (year)	a
Billion	B	Billion tonnes	Bt
British thermal unit	BTU	Centimetre	cm

Unit	Abbreviation
Cubic centimetre	cm^3
Cubic feet	ft^3
Cubic inch	in^3
Cubic yard	yd^3
Day	d
Days per year (annum)	d/a
Decibel	dB
Degree	°
Diameter	Ø
Dollar (Canadian)	C$
Foot	ft
Gallons per minute (US)	gpm
Gigapascal	GPa
Gram	g
Grams per tonne	g/t
Hectare (10,000 m^2)	ha
Horsepower	hp
Hours per day	h/d
Hours per year	h/a
Kilo (thousand)	k
Kilograms per cubic metre	kg/m^3
Kilograms per square metre	kg/m^2
Kilometres per hour	km/h
Kilotonne	kt
Kilovolt-ampere	kVA
Kilowatt hour	kWh
Kilowatt hours per year	kWh/a
Litre	L
Megabytes per second	Mb/sec
Megavolt-ampere	MVA
Metre	m
Metres Below Sea Level	mbsl
Metres per second	m/s
Microns	ųm
Milligrams per litre	mg/L
Millimetre	mm
Million bank cubic metres	Mbm^3
Minute (plane angle)	‘
Month	mo

Unit	Abbreviation
Cubic feet per minute	cfm
Cubic feet per second	ft^3/s
Cubic metre	m^3
Coefficients of variation	CVs
Days per week	d/wk
Dead weight tonnes	DWT
Decibel adjusted	dBa
Degrees Celsius	°C
Dollar (American)	US$
Dry metric ton	dmt
Gallon	gal
Gigajoule	GJ
Gigawatt	g
Grams per litre	g/L
Greater than	>
Hertz	Hz
Hour	h
Hours per week	h/wk
Inch	“
Kilogram	kg
Kilograms per hour	kg/h
Kilometre	km
Kilopascal	kPa
Kilovolt	kV
Kilowatt	kW
Kilowatt hours per tonne (metric ton)	kWh/t
Less than	<
Litres per minute	L/min
Megapascal	MPa
Megawatt	MW
Metres above sea level	masl
Metres per minute	m/min
Metric ton (tonne)	t
Milligram	mg
Millilitre	mL
Million	M
Million tonnes	Mt
Minute (time)	min
Ounce	oz

Unit	Abbreviation
Pascal	Pa
Parts per billion	ppB
Pound(s)	lb(s)
Revolutions per minute	rpm
Second (time)	sec
Square centimetre	cm^2
Square inch	in^2
Square metre	m^2
Three dimensional	3D
Tonnes per day	t/d
Tonnes per year (annum)	t/a
Total	T
Week	wk
Wet metric ton	wmt

Unit	Abbreviation
Parts per million	ppM
Percent	%
Pounds per square inch	psi
Second (plane angle)	"
Specific gravity	SG
Square foot	ft^2
Square kilometre	km^2
Thousand tonnes	kt
Tonne (1,000 kg)	t
Tonnes per hour	t/h
Tonnes seconds per hour metre cubed	ts/hm^3
Volt	V
Weight per weight	w/w

2.7 Terms of Reference (Abbreviations & Acronyms)

Table 2-5 shows Terms and Abbreviations used in this study. Table 2-6 shows the Conversions for Common Units.

Table 2-5: Terms of Reference

Unit	Abbreviation/Acronym
Absolute Relative Difference	ABRD
Abrasion Index	A_i
Acid Base Accounting	ABA
Acid Rock Drainage	ARD
Alpine Tundra	AT
Atomic Absorption Spectrophotometer	AAS
Atomic Absorption	AA
Bond Ball Mill Work Index	BW_i
Bond Rod Mill Work Index	RW_i
British Columbia	BC
British Columbia Environmental Assessment Act	BCEAA
British Columbia Environmental Assessment Office	BCEAO
British Columbia Environmental Assessment	BCEA
Canadian Dam Association	CDA
Canadian Environmental Assessment Act	CEA Act
Canadian Environmental Assessment Agency	CEA Agency
Canadian Institute of Mining, Metallurgy, and Petroleum	CIM
Canadian National Railway	CNR

Unit	Abbreviation/Acronym
Carbon-in-leach	CIL
Caterpillar's ® Fleet Production and Cost Analysis software	FPC
Closed-circuit Television	CCTV
Coefficient of Variation	CV
Copper	Cu
Copper Equivalent	CuEq
Counter-current decantation	CCD
Cyanide Soluble	CN
Digital Elevation Model	DEM
Direct Leach	DL
Distributed Control System	DCS
Drilling and Blasting	D&B
Environmental Management System	EMS
Flocculant	floc
Free Carrier	FCA
Gemcom International Inc.	Gemcom
General and Administration	G&A
Gold	Au
Gold Equivalent	AuEq
Heating, Ventilating, and Air Conditioning	HVAC
High Pressure Grinding Rolls	HPGR
Indicator Kriging	IK
Inductively Coupled Plasma	ICP
Inductively Coupled Plasma Atomic Emission Spectroscopy	ICP-AES
Inspectorate America Corp.	Inspectorate
Interior Cedar-Hemlock	ICH
Internal Rate of Return	IRR
International Congress on Large Dams	ICOLD
Invers Distance cubed	ID^3
Land and Resource Management Plan	LRMP
Lerchs-Grossman	LG
Life-of-Mine	LOM
Load-haul Dump	LHD
Locked Cycle Tests	LCTs
Loss on Ignition	LOI
Metal Mining Effluent Regulations	MMER
Methyl Isobutyl Carbinol	MIBC
Metres East	mE
Metres West	mW
Metres North	mN
Metres South	mS

Unit	Abbreviation/Acronym
Mineral Deposits Research Unit	MDRU
Mineral Titles Online	MTO
Nation Instrument 43-101	NI 43-101
Nearest Neighbour	NN
Net Invoice Value	NIV
Net Present Value	NPV
Net Smelter Price	NSP
Net Smelter Return	NSR
Neutralization Potential	NP
Northwest Transmission Line	NTL
Official Community Plans	OCPs
Operator Interface Station	OIS
Ordinary Kriging	OK
Organic Carbon	org
Potassium Amyl Xanthate	PAX
Predictive Ecosystem Mapping	PEM
Preliminary Assessment	PA
Preliminary Economic Assessment	PEA
Qualified Person	QP
Quality Assurance	QA
Quality Control	QC
Quality Assurance and Quality Control	QA/QC
Rhenium	Re
Rock Mass Rating	RMR
Rock Quality Designation	RQD
SAG Mill/Ball Mill/Pebble Crushing	SABC
Semi-autogenous Grinding	SAG
Silver	Ag
Silver Equivalent	AgEq
Standards Council of Canada	SCC
Stanford University Geostatistical Software Library	GSLIB
Tailings Storage Facility	TSF
Terrestrial Ecosystem Mapping	TEM
Total Dissolved Solids	TDS
Total Suspended Solids	TSS
Tunnel Boring Machine	TBM
Underflow	U/F
Valued Ecosystem Components	VECs
Waste Rock Facility	WRF
Water Balance Model	WBM
Work Breakdown Structure	WBS

Unit	Abbreviation/Acronym
Workplace Hazardous Materials Information System	WHMIS
X-ray Fluorescence Spectrometer	XRF

Table 2-6: Conversions for Common Units

Metric Unit	Imperial Measure
1 hectare	2.47 acres
1 metre	3.28 feet
1 kilometre	0.62 miles
1 gram	0.032 ounces (troy)
1 tonne	1.102 tons (short)
1 gram/tonne	0.029 ounces (troy)/ton (short)
1 tonne	2,204.62 pounds
Imperial Measure	**Metric Unit**
1 acre	0.4047 hectares
1 foot	0.3048 metres
1 mile	1.609 kilometres
1 ounce (troy)	31.1 grams
1 ton (short)	0.907 tonnes
1 ounce (troy)/ton (short)	34.28 grams/tonne
1 pound	0.00045 tonnes

3 RELIANCE ON OTHER EXPERTS

AGP has followed standard professional procedures in preparing the content of this Report. Data used in this report has been verified where possible, and this report is based upon information believed to be accurate at the time of completion considering the status of the Project and the purpose for which the report is prepared. AGP has no reason to believe the data was not collected in a professional manner.

AGP has not verified the legal status or legal title to any claims and the legality of any underlying agreements that may exist concerning the Property. Troilus supplied the list of mineral rights and mineral claim maps presented in this report. AGP has examined the Québec Ministère de l'Énergie et Ressources Naturelles (MERN) online GIS website (GESTIM) to correlate these mineral rights. The GESTIM website was most recently viewed on 20 July 2023 found here:

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

The QPs have also referenced several sources of information on the Property, including past reports by consultants to Troilus, digital geological maps, and other documents listed in the reference section of this report. Therefore, in authoring this report, the QPs have reviewed the work of the other contributors and find this work has been performed to normal and acceptable industry and professional standards.

3.1 Metallurgical Sample Selection

For the post-PEA metallurgical testwork program, Troilus selected a 3,000 kg bulk samples from each of the three deposits and based the on representativity using their domain and lithology. With the expansion of the resource in the FS, another set of 800 kg samples were collected based on a similar approach. As a verification of representativity of the sampling, Troilus retained an independent resource geologist to confirm the representativity of the samples collected for the testwork at Eriez. A memorandum detailing the findings was provided to Lycopodium for reference. The review concluded that the sample selection correctly targeted the areas that would form much of the reserve that will be ultimately mined, and the samples are representative of the tonnage distribution.

3.2 Metallurgical Testwork

Troilus retained Eriez to perform bench scale and pilot plant testwork using the three individual 3,000 kg samples for the post-PEA test program in 2021. Further testing was completed on the increased FS resource using the additional 800 kg selected from a second round of confirmatory testing at the Eriez lab. Both sets of representative samples returned similar performance and recoveries enabling Troilus and Lycopodium to confirm the performance of the process plant flow sheet developed from the Eriez test program. The test program was executed under the direction of Troilus.

3.3 Equipment Selection

The process design and equipment selection are based on the testwork with guidance from Eriez working in conjunction with Troilus and Lycopodium. Eriez is a world-renowned supplier of flotation equipment. As part of the equipment selection Troilus requested Eriez to provide an installation list of where the StackCells are currently being used so that a due diligence review could be completed. The due diligence included interviews with metallurgical staff from Red Chris Mine where similar sized Stack Cells were added to their existing rougher / scavenger circuit. Based on these interviews, Troilus is confident that the selected equipment will perform as per the specifications. Troilus recognizes that StackCells are considered new technology and there is a risk associated with using the equipment. This risk has been identified on the FS risk register. However, given Eriez is a world-renowned manufacturer of flotation equipment, and their reputation is at stake with the installation of the StackCells, Troilus believes Eriez has a vested interest in our Project and will provide all the required technical assistance necessary to guarantee its success.

In conclusion, Troilus and Lycopodium have performed an extensive amount of work to deliver a robust resource and metallurgical test program to support the process design and equipment selection.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location and Description

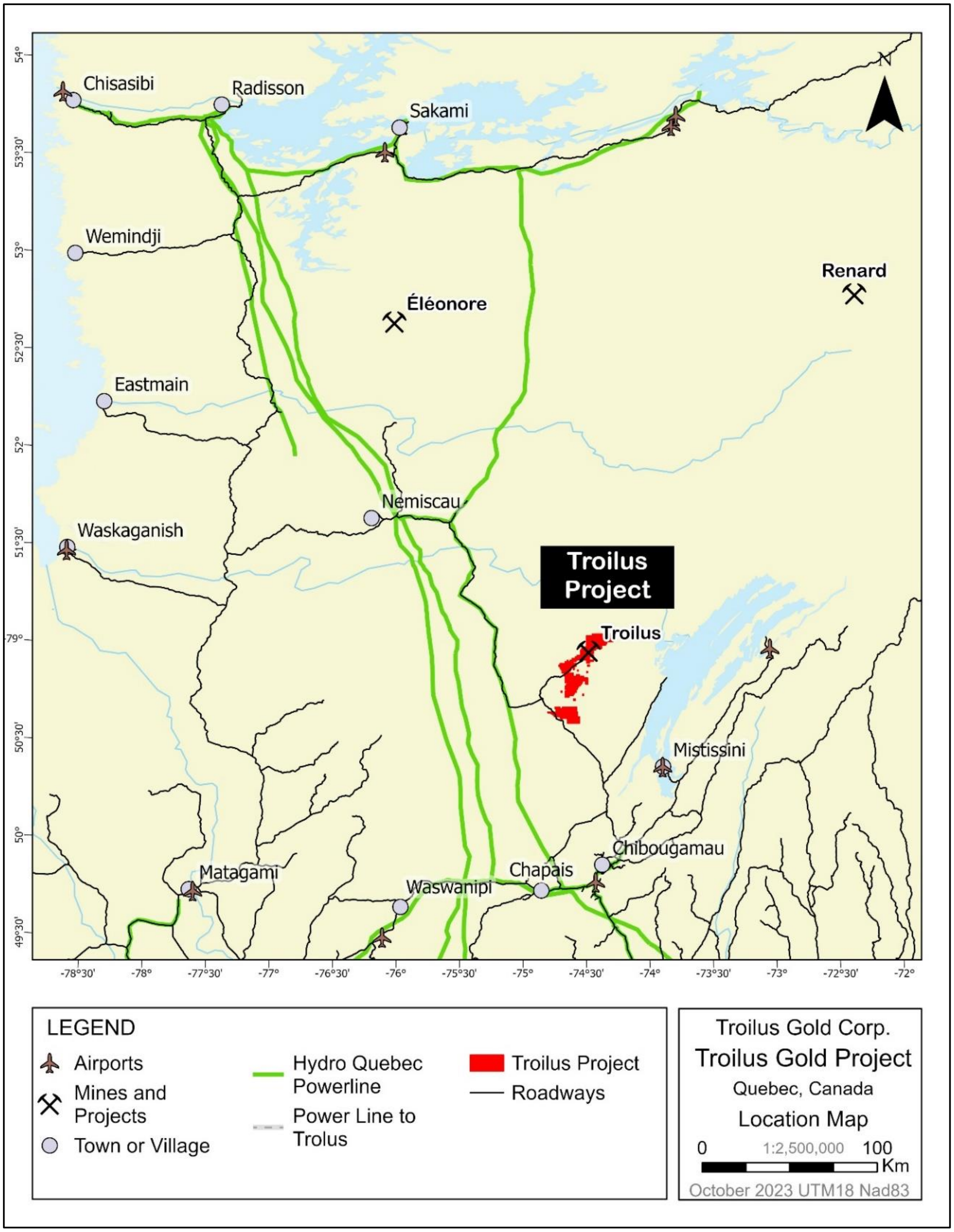
The Troilus Gold Property (Property) is divided into two projects: the Project and the Troilus Frotêt Project. The mineral rights to the Property cover a total area of approximately 44,124.88 ha. Of this total area, 7,242 ha is 50% owned by Troilus and 50% owned by Argonaut Gold through a joint venture (JV) agreement, with the remainder of the mineral rights being 100% held by Troilus.

The Property is located:

- on 1:250,000 scale Mapsheets NTS 023O (Lac Mesgouez) and 023J (Lac Assinica),
- on 1:50,000 scale Mapsheets 32J/15 (Lac Troilus), 32J/16 (Lac Bueil), 32O/01 (Lac Miskittenau), and 32O/02 (Lac Montmort),
- at approximately 51°00’ North and 74°30’ West,
- at approximately 538000 E; 4650400 N, Zone 18U (NAD83 datum) Universal Transverse Mercator (UTM) coordinates,
- at approximately 600 km north of Montreal,
- at approximately 170 km by road north of Chibougamau,
- in the Province of Québec,
- in the Administrative Region Nord-du-Québec,
- within the Wildlife Reserve (Réserve Faunique) Lacs Albanel Mistassini et Waconichi,
- approximately 45 km west of Lac Mistassini,
- approximately 9 km northeast of Lac Troilus

Figure 4-1 below shows the Property location in Québec.

Figure 4-1: Location Map, Central Québec



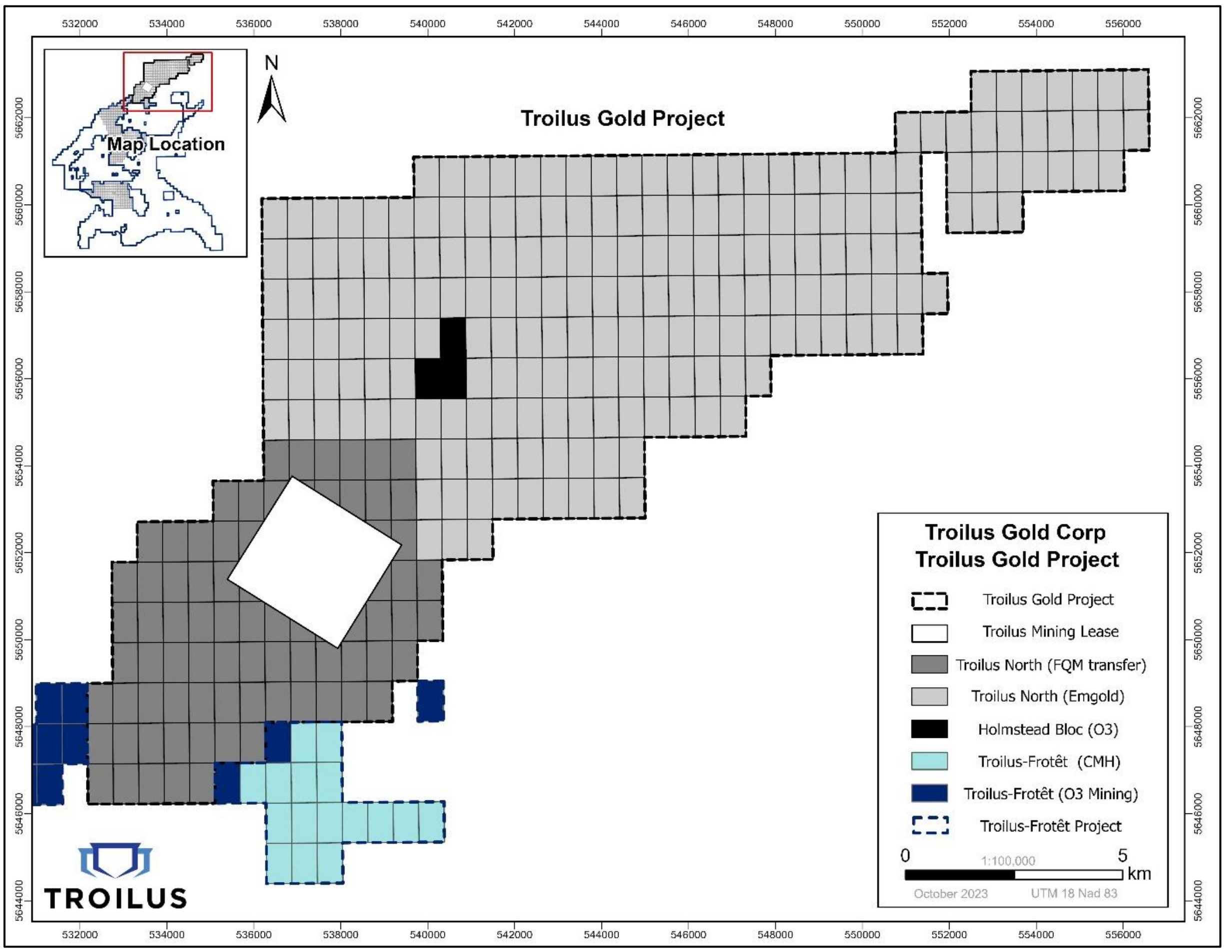
Source: Troilus (2023)

The mineral rights to the Project is comprised of a single Mining Lease (Bail Minier) and 293 mineral claims (Titres Miniers), totalling 16,185.09 ha. The mineral rights to the Troilus Frotêt Project is comprised of 520 mineral claims, totalling 27,939.79 ha. All mineral rights are in good standing.

Figure 4-2 presents the mineral rights map for the Project. Figure 4-3 presents the mineral rights map over both of the Projects.

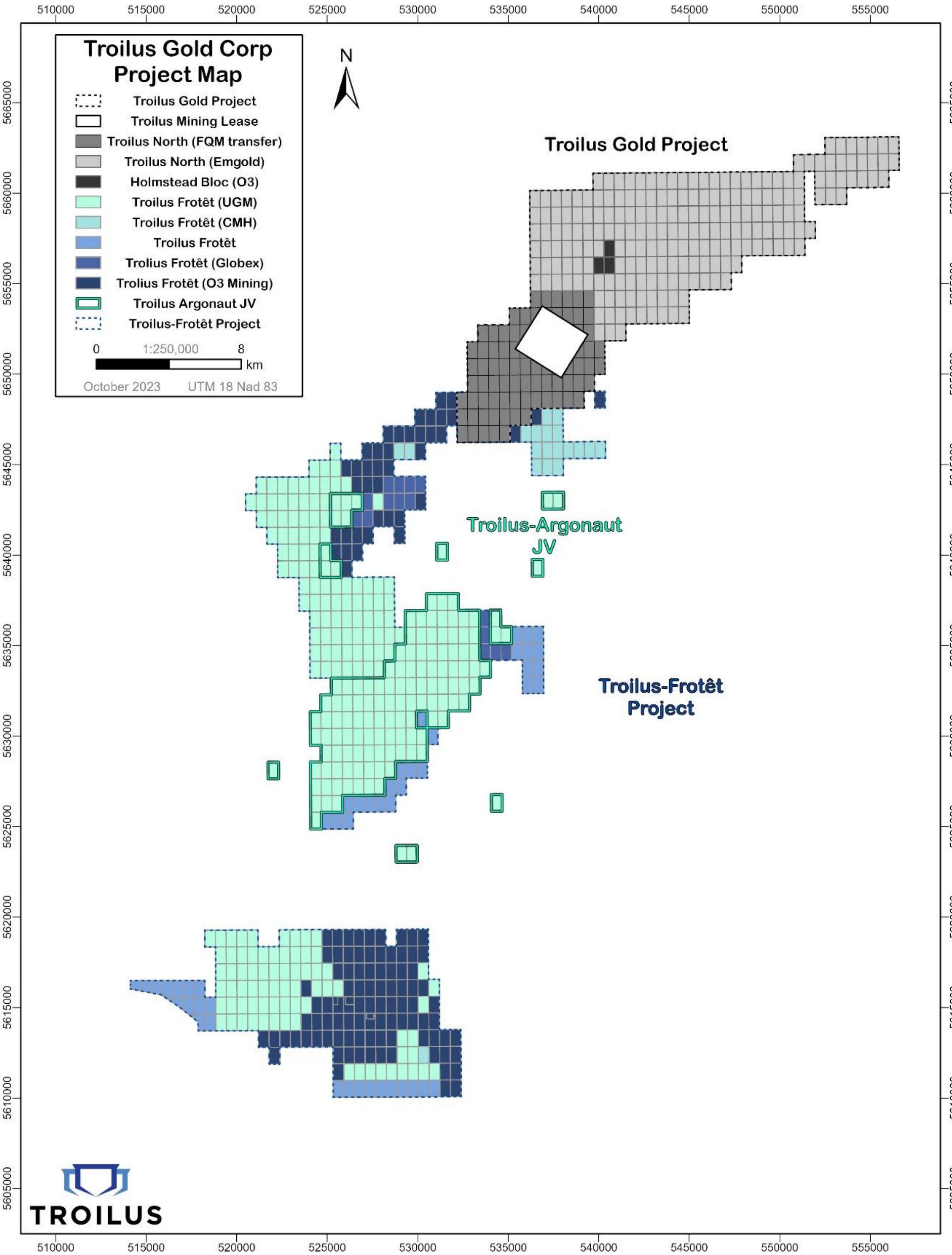
Table 4-1 lists the minerals rights for the Troilus Gold Property. Table 4-2 lists the mineral rights for the Troilus Frotêt Property.

Figure 4-2: Mineral Rights Map – Troilus Gold Project



Source: Troilus (2023)

Figure 4-3: Mineral Rights – Troilus Projects



Source: Troilus (2023)

Table 4-1: Summary of Mineral Rights for the Troilus Gold Property

Mineral Rights	Mineral Claim Number*	Count	Expiry Date	Area (ha)
Mining Lease (Bail Minier)	BM 829	1	Mar 2026	835.46
Mineral Claims Troilus Gold Project	2422145 – 2422147	3	Feb 2025	162.38
	2424713 – 2425732, 2424748 – 2424786, 2424958 – 2425037, 2488059	20 39 80 1	Mar 2025	7576.17
	1133905 – 1134008, 1133913 – 1133926, 1133929 – 1133930, 1133936 – 1133980, 1133982 – 1133985, 1133998 – 1134008, 2488138, 2488294 – 2488297	5 14 2 45 4 12 1 4	Apr 2025	4149.31
	2491523 – 2491527	5	May 2025	270.67
	2499212 – 2499223, 2500001 – 2500004	12 4	Aug 2025	865.30
	2502354 – 2502365	12	Sep 2025	648.80
	2504200 – 2504230	31	Oct 2025	1677.01
Subtotal Troilus Gold Project		**294**		**16,185.09**
Subtotal Troilus Frotêt Project		**520**		**27,939.79**
TOTAL		**814**		**44,124.88**

*list shows groupings of sequential mineral claim numbers

Table 4-2: Summary of Mineral Rights for the Troilus Frotêt Property

Mineral Rights	Mineral Claim Number*	Count	Expiry Date	Area (ha)
Mineral Claims (Troilus Frotêt Project)	2335740-2335741 2464724 -2464727 2543555-2543556 2543570-2543572 2543574-2543580 2543582 2543629-2543637 2543777-2543780	2 4 2 3 7 1 9 4	Sep 2026	1740.66
	2255903 2544110	1 1	Oct 2026	108.56
	2468129 2468134 2547409-2547418	1 1 10	Nov 2026	651.17
	2323689-2323692 2323697-2323698 2323700	4 2 1	Dec 2024	971.43

Mineral Rights	Mineral Claim Number*	Count	Expiry Date	Area (ha)
	2529164-2529169	6		
	2548537-2548539	3		
	2471376-2471379	4		
	2472351-2472355	5	Jan 2025	271.79
	1117913-1117917	5	Feb 2025	4518.32
	1117920-1117925	6		
	1117927-1117935	9		
	1117937-1117945	9		
	2513581-2513582	2		
	2513585-2513586	2		
	2513590	1		
	2513601-2513602	2		
	2513609-2513611	3		
	2555505-2555514	10		
	2555517-2555519	3		
	2555532-2555536	5		
	2555545-2555546	2		
	2555552-2555554	3		
	2555612-2555613	2		
	2555619-2555620	2		
	2555627-2555629	3		
	2555636-2555637	2		
	2555837-2555846	10		
	2556135-2556137	3		
	2424552	1	Mar 2025	1085.45
	2485089	1		
	2560654-2560655	2		
	2560662-2560663	2		
	2560668-2560669	2		
	2560673-2560681	9		
	2560806-2560808	3		
	1134209-1134213	5	Apr 2025	271.33
	2491363-2491374	12	May 2025	1739.99
	2492967-2492986	20		
	2498979-2498980	2	Jul 2025	163.02
	2499048	1		
	2336277-2336279	3	Augt 2025	1140.66
	2336281-2336286	6		
	2336288-2336290	3		
	2499783-2499791	9		
	2509780	1	Jan 2026	545.40
	2510218-2510219	2		
	2510277-2510279	3		
	2510296-2510298	3		
	2510302	1		
	2513583-2513584	2	Feb 2026	1581.52

Mineral Rights	Mineral Claim Number*	Count	Expiry Date	Area (ha)
	2513587-2513589	3		
	2513591-2513600	10		
	2513603-2513608	6		
	2513612-2513615	4		
	2531557-2531559	3		
	2532014	1		
	2534955-2534957	3	Mar 2026	491.26
	2535212-2535217	6		
	2515566-2515593	28	Apr 2026	2154.27
	2515595-2515602	8		
	2561837-2561841	5		
	2562429	1		
	2404418	1	May 2026	3320.80
	2517219-2517230	12		
	2517248-2517261	14		
	2517428-2517438	11		
	2517606-2517609	4		
	2517736-2517739	4		
	2518154-2518156	3		
	2539222	1		
	2539523-2539533	11		
	24497-24513	17	Jun 2026	3588.07
	2351879	1		
	2539742-2539743	2		
	2540669-2540670	2		
	2540576-2540578	3		
	2540922-2540923	2		
	2540925-2540938	14		
	2540955-2540958	4		
	2540993	1		
	2541129-2541134	6		
	2541166-2541172	7		
	2541191-2541193	3		
	2567480-2567484	4		
	2166908-2166910	3	July 2026	2508.39
	2166913	1		
	2166945	1		
	2166947-2166949	3		
	2334393	1		
	2334395-2334396	2		
	2335604	1		
	2541682-2541684	3		
	2541697-2541701	5		
	2541851-2541852	2		
	2454371-2454374	4		
	2454378-2454397	20		

Mineral Rights	Mineral Claim Number*	Count	Expiry Date	Area (ha)
	2454415	1		
	2542307-2542309 2542783 - 2542799 2456839-2456840	3 15 2	Aug 2026	1087.70
Subtotal Troilus Frotêt Project		**520**		**27,939.79**

*list shows groupings of sequential mineral claim numbers

4.2 Project Ownership

On May 2, 2016, a wholly owned subsidiary of Sulliden Mining Capital Inc., 2507868 Ontario Inc. (Sulliden Sub) entered into an agreement (agreement) with First Quantum Minerals Ltd. (First Quantum) to purchase a 100% interest in the Project, subject to a sliding scale NSR royalty. First Quantum had acquired the Troilus Mine as part of the takeover of Inmet Mining Corp. (Inmet) in March 2013.

To exercise the option under the Agreement, three cash payments of $100,000 were made to First Quantum and over $1,000,000 was spent by Troilus and its predecessors on engineering and technical studies to evaluate the economic viability of the Project. In addition, Troilus agreed to take on the existing liabilities of the Project.

On October 31, 2017, Pitchblack Resources Ltd. (Pitchblack), Sulliden Sub, and 2513924 Ontario Inc. (251 Ontario) entered into an amalgamation agreement. The amalgamation agreement closed on December 20, 2017, and Pitchblack was renamed Troilus.

Pursuant to the amalgamation agreement, Sulliden Sub, 251 Ontario, and a Pitchblack wholly owned subsidiary were amalgamated to form one wholly owned subsidiary of Pitchblack. Every four existing Pitchblack shares were consolidated into one new common share of Troilus.

On April 12, 2018, Troilus formally exercised its option to acquire the Property from First Quantum and title was transferred to Troilus. The 81 claims previously owned by First Quantum were previously subject to a variable NSR to First Quantum of 1.5% or 2.5% depending on whether the price of gold is above or below US$1,250 per ounce. This NSR was cancelled in a buy-back transaction that closed in November 2020. In addition, Sandstorm Gold Royalties has a 1% royalty on these 81 claims acquired through the acquisition of Nomad Royalty Company.

On December 5, 2018, Troilus announced that it had completed the acquisition of the Troilus North Project from Emgold. As consideration for the acquisition, Troilus issued Emgold 3,750,000 common shares and paid Emgold C$250,000 in cash.

The 209 claims acquired from Emgold Mining (formerly known as the Troilus North project) are subject to the following underlying royalties:

- 1% NSR to Emgold Gold Corporation that Troilus has the right to purchase for $1,000,000

On November 11, 2019, Troilus announced that it had completed the acquisition of three claims from O3 Mining Inc. (Holmstead Claims, Figure 4-2). As consideration for the acquisition of these three claims, Troilus issued 300,000 common shares and granted a 2% NSR to O3 Mining Inc. on these three claims. Troilus will have the right to repurchase 1% of the NSR at any time for $1,000,000. In addition, the three claims acquired from O3 Mining Inc. are subject to a 2% NSR to an individual, half of which can be purchased for $1,000,000.

On April 28, 2020, Troilus announced that it had completed the acquisition of a further 627 Claims from O3 Mining Inc. As consideration for the acquisition of the additional O3 Mining Inc. claims, the Company issued 1,700,000 common shares and granted a 2% NSR to O3 on the O3 Mining Inc. claims.

The majority of these claims have since been acquired by Sayona Mining Ltd. (Sayona) Troilus continues to hold 135 claims acquired from O3 Mining Inc. in April 2020 that are subject to the following royalties:

o 2% NSR to O3 Mining Inc., half of which can be purchased for $1,000,000, subject to the terms of the Buy Back agreement entered into between Troilus and Sayona Mining Ltd.

o 2% NSR granted to Inco Limited (now Vale) on seven of the 135 claims

On July 21, 2020, Troilus announced that it had completed the acquisition of 91 claims from Globex Mining Enterprises Inc. (Globex) as consideration for the acquisition of the Globex claims Troilus issued 350,000 common shares and granted a 2% Gross Metals Royalty (GMR) to Globex on the Globex claims. Troilus has the right to repurchase a 1% GMR on the Globex claims at any time for CAD$1,000,000. Troilus also announced that it had completed the acquisition of 21 claims from 9219-8845 Qc. Inc. dba Canadian Mining House (CMH). As consideration for the acquisition of the CMH claims Troilus paid cash consideration of CAD$69,000 and granted a 1% NSR to CMH on the CMH claims. Troilus has the right to repurchase a 0.5% NSR on the CMH claims at any time for C$500,000 and to purchase the remaining 0.5% NSR on at any time for CAD$1,500,000. The majority of the claims acquired from Globex and CMH have since been acquired by Sayona and the terms of the royalties above are subject to the terms of the Buy Back agreement with Sayona.

4.3 Mineral Tenure – Québec

In Québec, the Mining Act (Loi sur les mines) regulates the management of mineral resources and the granting of exploration rights for mineral substances during the exploration phase. It also deals with the granting of rights pertaining to the use of these substances during the mining phase. The Mining Act establishes the rights and obligations of the holders of mining rights to ensure maximum development of Québec's mineral resources (website: Québec Mining Act).

In Québec, mineral claims have pre-established positions and a legal survey is not required. A map designated claim is valid for two years and can be renewed indefinitely, subject to the completion of necessary expenditure requirements. The map designated mineral claims are approximately 54 ha but may be smaller due to where other rights supersede the claim. Each claim gives the holder the exclusive right to explore for mineral substances, except sand, gravel, clay, and other unconsolidated deposits, on the land subject to the claim. The claim also guarantees the holder's right to obtain an extraction right upon the discovery of a mineral deposit. Ownership of the mining rights confers the right to acquire the surface rights.

Mining Leases (Baux Miniers) are initially granted for a 20-year period. The mining lease can be renewed for additional ten-year periods.

4.4 Surface Rights

In addition to the surface rights covering the mining lease, there are surface right leases covering a number of areas with roads and infrastructure. The surface rights renewal fee for the mining lease totals more than $50,000 per year.

Troilus has complete access to all of the Property.

4.5 Royalties and Encumbrances

4.5.1 Royalties

The royalties specifically affecting the Project are presented below.

The 81 claims and BM 829 previously owned by First Quantum Minerals Inc. are subject to a 1% royalty to Sandstorm Gold Royalties acquired through the acquisition of Nomad Royalty Company.

The 209 claims acquired from Emgold Mining (formerly known as the Troilus North project) are subject to the following royalty:

- 1% NSR to Emgold Mining Corporation that the Company has the right to purchase for $1,000,000

The 3 claims acquired from O3 Mining Inc. in November 2019 are subject to the following royalties:

- 2% NSR to O3 Mining Inc., half of which can be purchased for $1,000,000
- 2% NSR to an individual, half of which can be purchased for $1,000,000

The 135 claims acquired from O3 Mining Inc. in April 2020 are subject to the following royalties:

- 2% NSR to O3 Mining Inc., half of which can be purchased for $1,000,000, subject to the terms of the Buy Back agreement entered into between Troilus and Sayona Mining Ltd.
- 2% NSR granted to Inco Limited (now Vale) on seven of the 135 claims

The 19 claims acquired from Canadian Mining House in July 2020 are subject to the following royalty:

- 1% NSR to Canadian Mining House, 0.5% of which can be purchased for $500,000 and 0.5% of which can be purchased by the Company for $1,500,000, subject to the terms of the Buy Back agreement entered into between Troilus and Sayona Mining Ltd.

The 15 claims acquired from Globex Mining Enterprises in July 2020 are subject to the following royalty:

- 2% GMR (Gross Metal Sales) to Globex Mining Enterprises, 1% of which can be purchased by the Company at any time for $1,000,000, subject to the terms of the Buy Back agreement entered into between Troilus and Sayona Mining Ltd.

The Bullseye claims acquired through the acquisition of UGM that are subject to a 50% joint venture agreement with Argonaut Gold are subject to the following royalties:

- 13 claims in NTS 032J15 totaling 704.34 hectares are subject to a 2% NSR to O3, half of which can be purchased at any time for $500,000. UGM acquired the claims from O3

Under the joint venture agreement with Argonaut, in the event that either party's participating interest is diluted to 10% or less (a "Diluted Participant"), the other party shall have the right to cause the Joint Venture to redeem the participating interest held by the Diluted Participant in exchange for a royalty interest equal to 2% net smelter return ("NSR") royalty, half of which can be purchased from the date of issue of the NSR for $1,000,000.

The 100% owned Pallador claims acquired through the acquisition of UGM are subject to the following royalties:

- Seventy-one (71) claims, in NTS 032J15 totaling 4,182.33 hectares, on the Dileo-Nord property acquired through the UGM amalgamation are subject to a 1% NSR royalty to Soquem half of which can be purchased at any time for $500,000. UGM acquired the claims from Soquem, subject to the terms of the Buy Back agreement entered into between Troilus and Sayona Mining Ltd.
- 55 claims totaling 2,999.31 hectares in NTS 32J10, acquired through the UGM amalgamation are subject to a 1% NSR to Geotest Corporation (0.5%) and Wayne Holmstead (0.5%). UGM acquired the claims from Geotest/Holmstead

4.5.2 Encumbrances

In 2007, the site restoration work began by Inmet with the re-vegetation of areas no longer used by Troilus (Figure 4-4 to Figure 4-7). The dismantling, cleaning, and grading work has largely been completed. Fertilization and seeding work is on-going, particularly in the tailings area. A water treatment plant has been functional since the end of 1998, after initial operation revealed suspended solid control problems. It uses a new technology (ACTIFLO) based on polymer addition and agitation followed by high-speed sand assisted lamellar decantation and reduces suspended solids to concentrations below 15 ppm, the monthly average regulation limit. The length of time the water treatment plant will be required for is unclear.

The first version of the mine restoration plan was filed with the Ministère des Ressources Naturelles et de la Faune (MRNF) in 1996, followed by a first revision in 2002 and a second revision five years later in 2007.

The current mine restoration plan was produced by Genivar Inc. (Genivar) in November 2009 (Genivar, 2009). This restoration plan took into consideration the previous versions, however, was a completely new plan including the recent additional studies updating the information regarding the hydrology and hydrogeology, the acid rock drainage, the Phase 1-type site characterization, and the progressive restoration work carried out in 2007, 2008, and 2009. The Cree Nation of Mistissini (the Mistissini Cree) community was consulted throughout the process. The closure plan for the Troilus Mine was approved by the Québec Ministry of Sustainable Development, Environment and Parks (Certificate of Authorization No. 3214-14-025) pursuant to modifications made 3 November 2010 and 23 May 2012.

Surface and groundwater water samples are taken at regular intervals at a number of different monitoring sites on the property and annual reports summarizing the results are submitted to the MRNF and the Ministère de l'Environnement et de la Faune (MDDEP).

Genivar (2009) estimated that the site restoration work would be completed in 2012 and that the post-restoration monitoring program would continue until 2016. AGP notes that the site restoration work is ongoing and may take longer than anticipated. AGP recommends that Troilus re-assess the timing and costs related to site restoration and monitoring and recommends an environmental expert be retained to review ongoing monitoring and site restoration work.

Figure 4-4: Troilus Z87 (foreground) and J Open Pits and Waste Dumps (looking northwest)

Source: Troilus (2023)

Figure 4-5: Troilus Z87 Zone Open Pit (looking south)

Source: Troilus (2023)

Figure 4-6: Troilus J Zone Open Pit (looking north)

Source: Troilus (2023)

Figure 4-7: Troilus J Zone Open Pit (looking northwest)

Source: Troilus (2023)

4.6 Permits

No permits are required to conduct exploration activities on the Property other than a permit for tree cutting pertaining to the installation of drill roads and drill setups. The permit for tree cutting is issued by the Ministère des Forêts, de la Faune et de Parcs (MFFP).

4.7 Environmental Liabilities

AGP is unaware of any environmental liabilities or other factors and risks that may affect access, title or ability that would prevent Troilus from conducting exploration activities on the Property.

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

5.1 Accessibility

The Project is located 170 km by road, north of Chibougamau. From Chibougamau, the Property is easily accessed by driving 23 km east and northeast along 3e Rue and Highway 167, turning north on Route du Nord for approximately 108 km: and turning east and northeast along the mine access road (R1047) for roughly 44 km. Highway 167 is paved and in good condition. The Route du Nord and mine access road are well maintained year-round. The drive from Chibougamau is typically 2 hours.

There are regular scheduled flights to Chibougamau from Montreal.

5.2 Climate

The region where the Property is situated has a Continental Subarctic climate (Dfc; Köppen climate classification) characterized by long cold winters and short mild summers. Mean temperatures range from -20°C in January to 16°C in July. Mean annual precipitation ranges from 51 mm in February to 106 mm in August (Mistissini; worldclimate.com).

Exploration and mining activities may be carried out all year round.

5.3 Local Resources and Infrastructure

The nearest town to the Property is Mistissini, a Cree community located approximately 90 km southeast of the site. There are limited services available at Mistissini. In June 2018, Troilus opened an office at Mistissini that provides a forum for exchanging information and liaising with the Cree on a variety of social, environmental, and economic aspects of the Project, and the potential for future training, employment, and business opportunities.

Chibougamau, with a population approximately 7,500 (est. 2016), is the largest town in Nord-du-Québec, and offers most services, supplies and fuel required for the Project. Chibougamau is a well-established mining town and has a well-developed local infrastructure, services, and a mining industry workforce. In October 2018, Troilus opened an exploration office in Chibougamau.

The Property is connected to the provincial hydroelectric grid via a 137 km 161 kV power line. Water on the Property is abundant and available for exploration activities.

Politically, the Québec province is very supportive of mining. The Québec government has demonstrated a will to encourage the development of natural resources through expeditious permitting, title security, and financial incentives.

Troilus maintains local infrastructure around the historic mine site. The key current infrastructure includes:

- an 80-person camp; accommodation and kitchen

- exploration office building
- core logging and sampling facility
- outdoor core storage area
- garage for snow removal and road maintenance contractor
- garage for site restoration employees
- electrical transformer station
- drinking water tank and pump house
- tailings water treatment plant
- tailings storage facility
- waste management facilities
- several tailings water pump houses
- gatehouse and gate

5.4 Physiography

The Project area is primarily covered by black spruce forests, swamps, and lakes. The vertical relief in the area is moderate, between 370 m and 500 mASL. The historic Troilus Mine is situated on the western flank of a 500 m tall hill at a mean altitude of 375 mASL. Overburden consists of a thick layer (greater than 10 m) of fluvio-glacial till. Outcrops are sparse and very large boulders sitting on surface are common.

In addition to the surface rights covering the mining lease, there are surface rights leases covering several areas with roads and infrastructure. The extent of the surface rights was sufficient to operate the mine in the past, however, additional surface rights may be required as mineral resources are added to the current Project.

5.5 Sufficiency of Surface Rights

Troilus has sufficient surface rights to access and conduct exploration activities on the Property.

6 HISTORY

Initial exploration in the area began in 1958 following the discovery of many erratic blocks containing copper and nickel anomalies. Some occurrences of copper and zinc were discovered between 1958 and 1967, including a massive sulphide deposit at Baie Moléon discovered by Falconbridge Ltd. in 1961.

In 1971, the Lessard deposit was discovered by Selco Mining Corp. near Lac Domergue. It was geologically similar to Baie Moléon, consisting of massive sulphides. Following this discovery, an electromagnetic (EM) and magnetic geophysical survey was carried out over the Troilus and Frôtet Lake area; however, this survey did not lead to any new significant discoveries.

The Baie Moléon and Lessard discoveries, located southwest of the Troilus deposit, improved the geological understanding of the Frotêt-Evans greenstone belt, and opened the area to further exploration for base metal deposits.

In 1983, the results of a new airborne INPUT survey carried out over a large area of the eastern portion of the Frôtet-Evans belt were published by the Government of Québec. Some exploration work was conducted following this survey; however, no important discoveries were made.

6.1 Exploration and Development (1985 – 2010)

Table 6-1 presents a summary of the exploration and development history of the Troilus Mine from 1985 to 2010.

Table 6-1: Summary of the history of the Troilus Mine, 1985 – 2010

Date	Description
1985	Kerr Addison Mines Ltd. (Kerr Addison) stakes over 1,500 claims in the Troilus area
1987	Kerr Addison stakes Troilus Mine area and discovers gold and copper
1988	Minnova Inc. (Minnova) options 50% interest from Kerr Addison and becomes operator
December 1991	Kilborn Inc Pre-Feasibility Study is negative (7,500 t/d)
Feb to May 1993	Metall Mining Corporation (Metall) acquires 100% interest in Troilus
August 1993	Kilborn-Met-Chem-Pellemon Feasibility Study is positive (10,000 tpd)
September 1994	Metallgesellschaft AG sold its entire 50.1% interest in Metall through the public sale of its shares
Late 1994	Construction commenced
May 4, 1995	Metall changed its name to Inmet Mining Corp (Inmet)
1995	44 km access road from Route du Nord and a 137 km power line and two substations were completed
October 1996	Construction completed
November 1996	Production at the Troilus Mine starts
April 1997	Mill achieves 10,000 t/d
April 1998	Met-Chem 15,000 t/d mill expansion Feasibility accepted
1999	Mill achieves 15,000 t/d
2002	Mill achieves 16,000 t/d
2004	Met-Chem 20,000 t/d mill expansion Feasibility accepted
2005	Mill achieves 20,000 t/d
2007	Underground ramp stopped at 519.1 m from portal on 22 January 2007
2008	Mining at J4 pit (now J Zone pit) completed in May 2008
2008	Dumping waste backfill at south end of J4 pit begins in April 2008.
2009	Mining at Z87 pit completed, last truck load on 13 April 2009
2010	Mill stopped on 29 June 2010
2010	Mill sold and shipped to Mexico in September 2010
2010	Camp sold on 19 November 2010 and subsequently dismantled

6.1.1 Ownership History (1985 – 1993)

Kerr Addison staked two large blocks of claims in 1985 and 1987 that included the Project area. In 1988, Minnova became operator in a 50%-50% joint-venture with Kerr Addison.

In February 1993, Metall acquired Minnova's interest and, in May 1993, Metall purchased all of Kerr Addison's mining properties. On May 4, 1995, Metall changed its name to Inmet.

Inmet was acquired by First Quantum in March 2013. On 8 April 2014, Copper One entered into a definitive purchase agreement with FQM (Akubra) Inc., a wholly owned subsidiary of First Quantum, to acquire a 100% interest in the past producing Troilus Mine, however, the purchase was not completed.

6.1.2 Kerr-Addison and Minnova (1985 – 1993)

In 1985, Kerr Addison acquired a large block of claims following a geological mapping program by the Québec Ministry of Natural Resources that indicated good potential for gold and base metal mineralization. More geochemical, geophysical, and geological work was carried out by Kerr Addison in 1985 and 1986. Drilling began in 1986 with 24 holes totalling 3,590 m, which led to the discovery of Zone 86 (Z86).

In 1987, more claims were added to the property to the north of the Z86 drilling, where the former Troilus Mine is currently located. A large gold float dispersion train was found by prospecting and 26 diamond drill holes totalling 4,413 m were completed. Hole KN-12, collared immediately up-ice from a glacial float dispersion train, intersected significant gold-copper mineralization over great widths, which turned out to be part of Z87, named after the year of its discovery.

In 1988, 27 diamond drill holes totalling 6,567 m were completed. Initial drill testing of a nearby weak horizontal loop electromagnetic (HEM) anomaly intersected anomalous gold-copper mineralization in what was later confirmed to be J4 in 1991. The J4 name originates from its location on the "J" exploration grid. On October 1, 1988, a 50-50 joint-venture was formed between Kerr Addison and Minnova. Minnova became the operator.

Between 1989 and 2005, fourteen drilling programs comprising 887 diamond drill holes for a total of 159,538 m were carried out on the property. The drilling outlined five main areas of gold mineralization (Z87/Zone 87 South (Z87S), Z87 Deep, J4, J5, and Southwest (SW)), and several isolated gold intersections.

In 1991, a semi-permanent camp, which could accommodate 30 to 50 people, was set up between Z87 and J4. During 1991, a bulk sample of approximately 200 t, averaging 2.3 g/t Au was taken from the centre of Z87 and approximately 100 t were treated at the pilot plant of the Centre de Recherche Minérale du Québec in Québec City as part of a pre-feasibility study. The remaining 100 t were treated at the pilot plant of SGS Lakefield Research Limited (Lakefield) as part of the 1993 feasibility study.

In 1992, an orientation Induced Polarization Survey (IP) carried out over Z87 and J4 produced strong IP anomalies. The IP survey covered the entire Property and was also useful in planning of a condemnation drilling program in areas where the infrastructure and stockpiles were planned.

Between December 1992 and March 1993, a drilling program comprising 181 holes totalling 24,239 m was carried out to complete the feasibility study. The purpose of the drilling was to define Z87 and J4 as well as to test other IP anomalies.

6.1.3 Metall and Inmet (1993 – 2005)

In February 1993, Metall acquired Minnova's interest and, in May 1993, purchased all of Kerr Addison's mining property interests. In August 1993, a positive feasibility study was completed based on a 10,000 t/d open pit operation (Kilborn, 1993). In September 1993, the Coopers & Lybrand Consulting Group from Toronto, Ontario, audited the feasibility study and found no significant problems.

From August 1994 to April 1995, Mineral Resources Development Inc. (MRDI) from San Mateo, California, reviewed the reserves of both the feasibility and post-feasibility studies for financing purposes. Other kriging parameters were tested, and a check assay program was carried out on the 1992 to 1993 data set.

In May 1995, Metall changed its name to Inmet. Financing of the project was completed in June 1995. Later that year, the refurbishing of the 44 km access road from the Route du Nord and a 137 km power line and two substations were completed.

The construction of the mill complex, and all facilities was completed in the fall of 1996, and milling started in November 1996. In April 1997, after some fine tuning, the mill capacity reached 10,000 t/d.

In April 1998, Inmet approved a 15,000 t/d mill expansion feasibility study by Met-Chem Canada Inc. (Met-Chem). Modifications to the mill started in December 1998, and the full 15,000 t/d capacity was achieved in 1999.

New sampling and assay protocols for the blastholes and future diamond drilling campaigns were proposed by Francis Pitard in January 1999 (Pitard, 1999). As a result, significant modifications to the Troilus assay laboratory were completed during the fall of 1999 and it became fully operational in May 2000, after a six-month implementation and adjustment period.

In 2004, Inmet approved another mill expansion feasibility study by Met-Chem to increase mill capacity to 20,000 t/d. Modifications to the mill were completed in December 2004 and the full 20,000 t/d capacity was reached in 2005. In 2010, the mine was shut down as Inmet’s direction shifted to other assets.

6.2 Historic Production - Troilus Mine (1996 – 2010)

The Troilus Mine was a conventional open pit that operated on a continuous, year-round basis. The mill had a nominal capacity of 20,000 t/d with a flow sheet consisting of a gravity recovery and flotation circuit. There was a permanent on-site camp with dining, sleeping, and recreational facilities for up to 450 workers, which has since been dismantled. Security personnel patrolled the site on a regular basis. When the former Troilus Mine was in operation, bus transportation was provided for the workforce several times per week to and from Chibougamau and Mistissini.

The mine started commercial production in October 1996 and operated continuously up to April 2009 and the mill continued to process stockpile material up to 29 June 2010.

From 1995 to 2010, approximately 69.6 Mt averaging 1.00 g/t Au and 0.10% Cu of ore was mined and 7.6 Mt of lower grade mineralization had been stockpiled. A total of approximately 230.4 Mt had been excavated including 18.4 Mt of overburden and 134.7 Mt of waste rock.

The overall mill recovery averaged 83% for gold and 89% for copper. The Troilus Mine produced over two million oz of gold and almost 70,000 t of copper. The mill processed the low-grade stockpile material from 2009 up until June 29, 2010. The production history up to the end of the mine life in 2010 is summarized in Table 6-2.

Table 6-2: Historical Production, Troilus Mine, 1996 – 2010

Description	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004	2005	2006	2007	2008	2009	2010	1995 - 2010
Overburden (000 t)	3,449	5,080	3,235	967	1,949	552	63	203	843	1,702	347	0	0	0	0	0	18,390
Waste Rock (000 t)		988	8,840	13,052	12,073	14,370	13,441	14,912	11,279	10,344	11,452	9,787	6,951	6,999	212	0	134,700
Stockpile (000 t)		118	865	1,423	1,144	61	1,081	8	261	468	888	371	167	784	0	0	7,639
Ore Mined (000 t)		629	3,798	4,176	4,959	4,913	5,901	5,943	5,923	6,045	6,929	6,670	6,463	5,599	1,692	0	69,640
Total Excavated (000t)	3,449	6,814	16,737	19,618	20,126	19,895	20,485	21,065	18,307	18,559	19,616	16,828	13,582	13,382	1,904	0	230,367
Mill Head (g/t Au)		1.35	1.44	1.34	1.26	0.9	1.1	1.08	1.03	0.95	0.94	0.86	0.87	0.95	0.83	0.52	1.03
Mill Head (%Cu)		0.157	0.163	0.138	0.125	0.104	0.156	0.132	0.108	0.092	0.076	0.051	0.054	0.106	0.11	0.08	0.11
Gold Recovery		80.7	85.56	86.43	85.64	82.78	83.6	83.05	83.01	80.63	81.79	82.45	81.72	84.02	84.00	81.00	83.09
Copper Recovery		81.4	89.41	89.71	89.81	89.87	91.75	90.22	89.42	86.78	89.68	86.9	87.63	93.39	92.00	89.00	89.13
Au (oz)*		12,941	139,888	146,970	168,364	122,532	162,578	164,602	164,061	149,028	159,545	147,876	138,391	151,297	135,200	37,900	2,001,173
Cu (t)*		471	5,158	4,915	5,416	4,786	7,836	6,817	5,791	4,814	4,444	2,881	2,772	5,707	5,900	2,000	69,708

Note: *Recovered metal after milling and smelter and refining adjustments

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

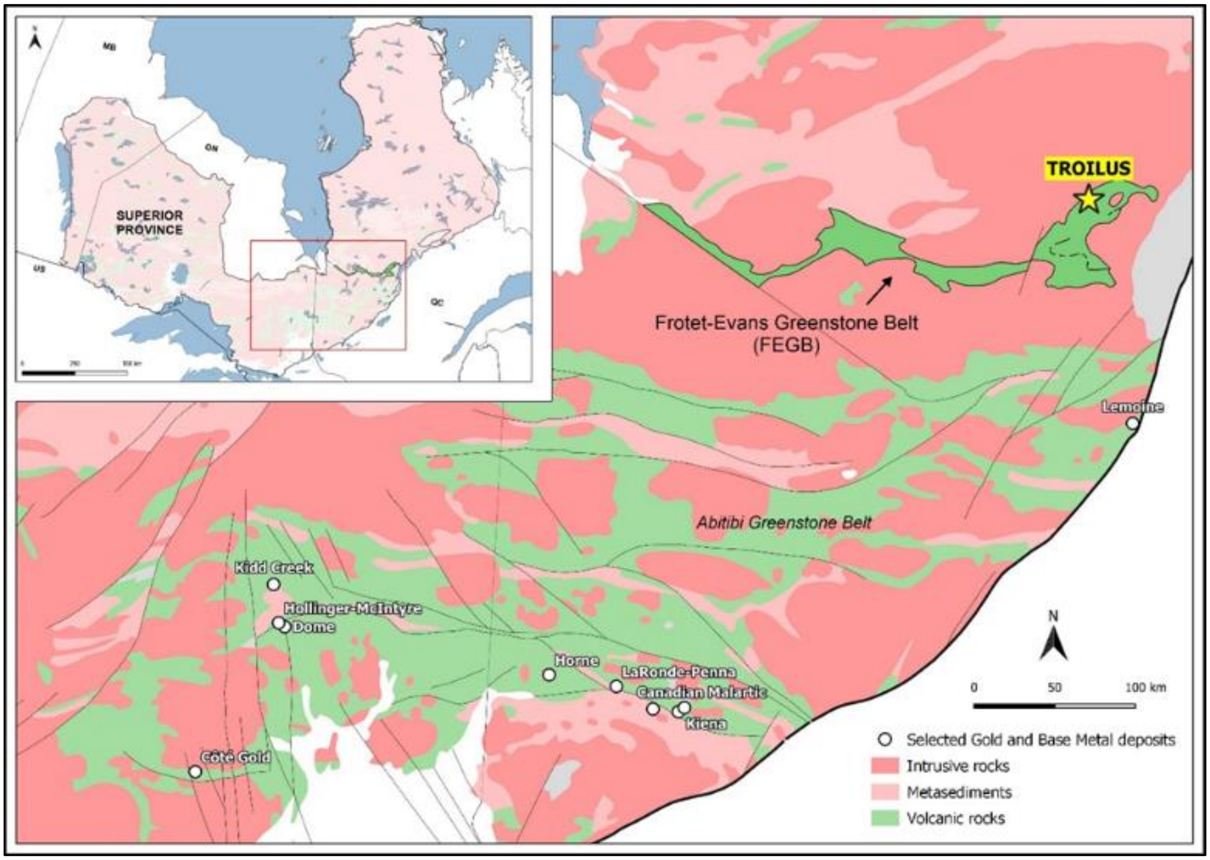
The Troilus Gold deposit lies within the eastern segment of the Frotêt-Evans Greenstone Belt (FEGB), in the Opatica Subprovince of the Superior Province in Québec. Figure 7-1 presents the Regional Geology of the FEGB.

The FEGB is centrally located in the Opatica Subprovince and extends for 300 km between James Bay, in the west, and Lake Mistissini, in the east, with variable widths, up to 45 km in its eastern extents (Carles, 2000). Its volcanic rocks define an east-west, fault-bounded trending synformal structure (Simard, 1987; Davis et al., 1995). The FEGB volcano-sedimentary sequence can be broadly divided in two similar domains, west and east. Detailed subdivisions have been made by Brisson et al., (1997a, b and 1998a, b, c), and Morin (1998 a, b, c) in a series of geological mapping initiatives developed throughout the greenstone belt by the Ministry of Natural Resources of Québec. Boily and Dion (2002) divided the FEGB in four distinctive segments: (1) Evans-Ouagama, (2) Storm-Evans, (3) Assinica, and (4) Frotêt-Troilus. Figure 7-2 shows the eastern Frotêt-Troilus domain (Simard, 1987) and has received primary focus for development due to its larger economic potential.

The FEGB is largely dominated by tholeiitic basalts and magnesian basalts that occur in association with felsic and intermediate calc-alkaline pyroclastic rocks, lava flows, and local ultramafic layers. Syn- to post-deformational gabbroic to monzogranitic plutonic rocks occur throughout the greenstone belt.

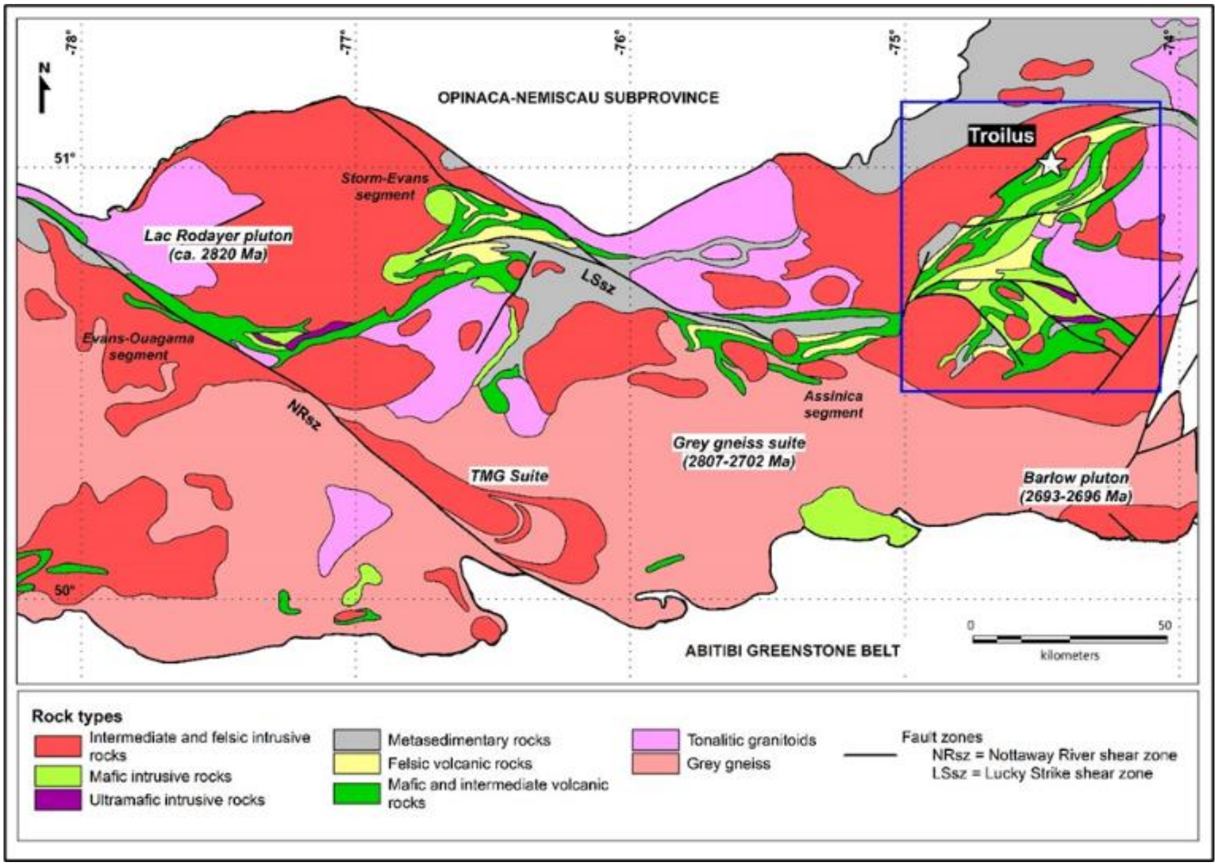
The few published U-Pb dates in zircon constrained the age of the FEGB between 2793 Ma and 2755 Ma (Pilote et al., 1997 in Boily and Dion, 2002). The circa 2793 Ma age is coincident with the dates obtained for the Troilus diorite.

Figure 7-1: Regional Geology Map – Frotêt-Evans Greenstone Belt



Source: Troilus (2019)

Figure 7-2: Regional Geology Map – FEGB in Central Québec



Source: Troilus (2019)

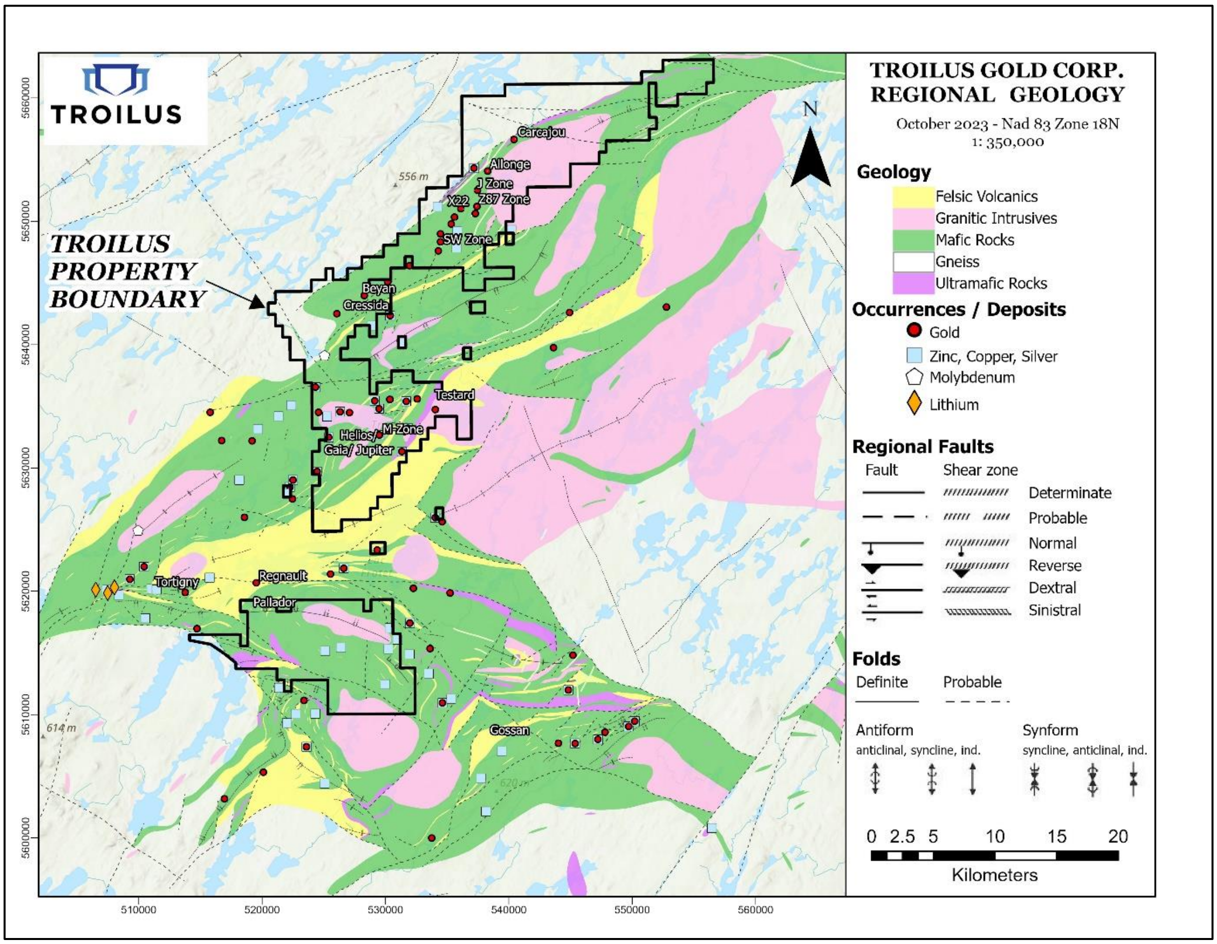
The Troilus Gold Deposit is situated in the Frotêt-Troilus domain in the east of the FEGB (Figure 7-3). It is characterized by a complex and variable volcano-magmatic history, dominated by mafic volcanic rocks and coeval, cogenetic mafic intrusions, intermediate to felsic volcanic rocks and associated pyroclastic rocks. Minor epiclastic sedimentary rocks and ultramafic units are locally observed.

The domain is divided in two structural regions, north and south, with the limit between them defined by the axial trace of the Frotêt Anticline (approximately E-W direction). The rocks are variably deformed and are affected by a strong regional foliation. Sub horizontal mesoscopic to megascopic folds are common, affecting both regional foliation and primary layering. The main regional structures observed in the northern structural domain are: (i) Troilus Syncline; (ii) La Fourche and Dionne dextral fault zones; and (iii) Parker inverse fault zones (Gosselin, 1996). The Troilus deposit is hosted in the northern overturned limb of the Troilus syncline. The Troilus syncline is characterized as an isoclinal fold of northeast-southwest strike. The associated axial plane is parallel to the main foliation in the region, which strikes northeast and has a moderate to steep dip towards the northwest (Fraser, 1993). The La Fourche and Dionne fault zones locally cut and segment the Troilus Syncline and correspond to important deformation corridors with an interpreted dextral sense movement. They are characterized by local centimetric to metre-scale isoclinal folds that affect the main regional schistosity, forming a crenulation cleavage. A locally pronounced, sub horizontal stretching lineation can be observed in places. The Parker fault zones represent a complex array of inverse faults, which are oriented predominantly parallel to bedding and the main regional foliation. The southern domain shows a more complex structural style with a series of major folding systems cut by several fault zones. Faults, axial fold planes and the main schistosity have an overall west-northwest- east-southeast to northwest-southeast direction.

The regional metamorphic grade in the Troilus area varies from greenschist facies in the internal sectors of the belt to lower-amphibolite facies near the felsic intrusions and the borders of the belt (Gosselin, 1996). The higher metamorphic grade is apparent adjacent to boundaries of intrusions and margins of the greenstone belt.

The Troilus region contains many occurrences of gold, base metal, and molybdenite mineralization, with the Troilus gold deposit being the largest. The three largest base metal volcanogenic massive sulphide (VMS) occurrences are the Lessard, Tortigny, and Baie Moleon deposits.

Figure 7-3: Regional Geology Map – Troilus Gold Project in the FEGB



Source: Troilus (2023)

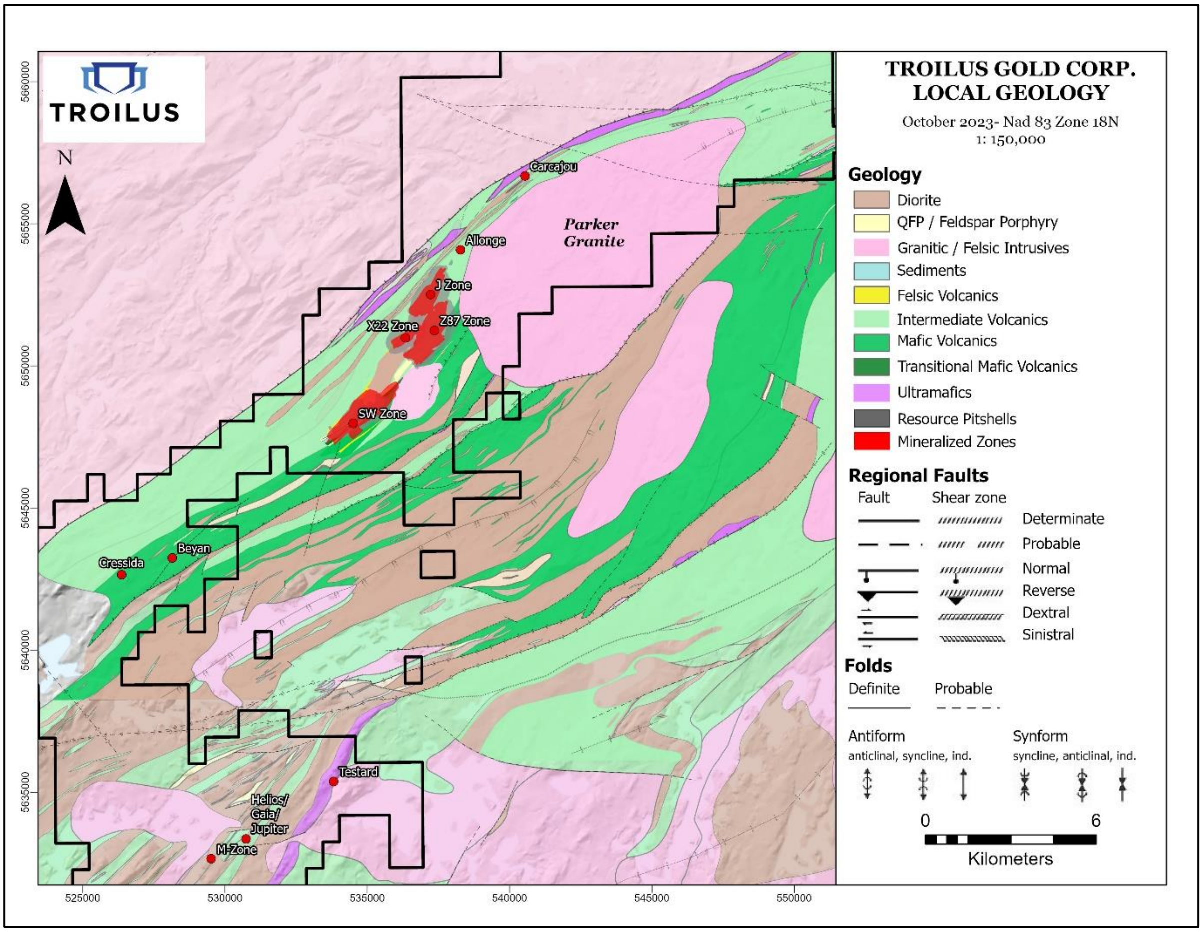
7.2 Project Geology

The Troilus deposit is located in the northeastern region of the Frotêt-Troilus domain and is hosted by volcanic and hypabyssal intrusive rocks of the Troilus Group in a region of intense deformation, known as the Parker domain (Gosselin, 1996). It is located within the overturned northern limb of the Troilus isoclinal syncline, which was transposed by a series of northeast to southwest striking thrust fault zones, parallel to the main regional foliation and to the volcanic bedding.

The Troilus Group on the Property is represented by a thick volcanic sequence, predominantly mafic to intermediate in composition. Synvolcanic magmatism is marked by a series of gabbro and ultramafic sills (Figure 7-4). Figure 7-5 shows the main lithotypes which comprise the Troilus deposit region are a metadioritic pluton with brecciated margins mafic to intermediate flows and volcanoclastic rocks, which are crosscut by multiple generations of felsic dykes. Late-stage dykes of mafic composition and syn to post-tectonic granitic plutons crosscut all these rock types. The lithological contacts and a penetrative foliation steeply dip to the northwest.

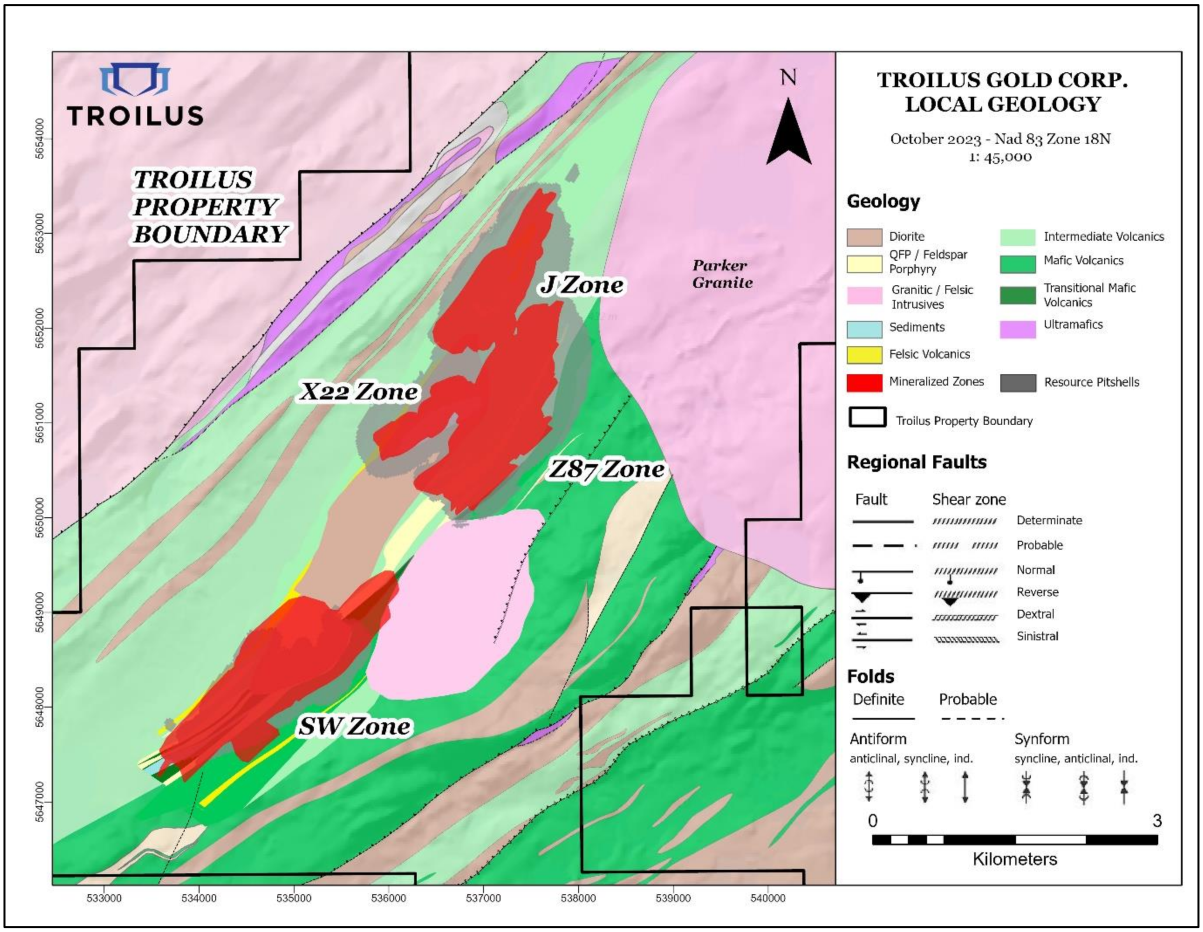
The following descriptions for the main lithologies, alteration, mineralization, and structural features are based mostly on descriptions and lithogeochemical studies of diamond drill holes drilled from 2018 to 2022 by Troilus Gold, as well as contributions from the works of Brassard (2018), Brassard & Hylands (2019), Diniz (2019), Laurentia Exploration (2018), and SRK (2018)

Figure 7-4: Property Geology Map – Troilus Gold Project



Source: Troilus (2023)

Figure 7-5: Property Geology Map – Troilus Gold Project



Source: Troilus (2023)

7.2.1 Deposit Lithologies

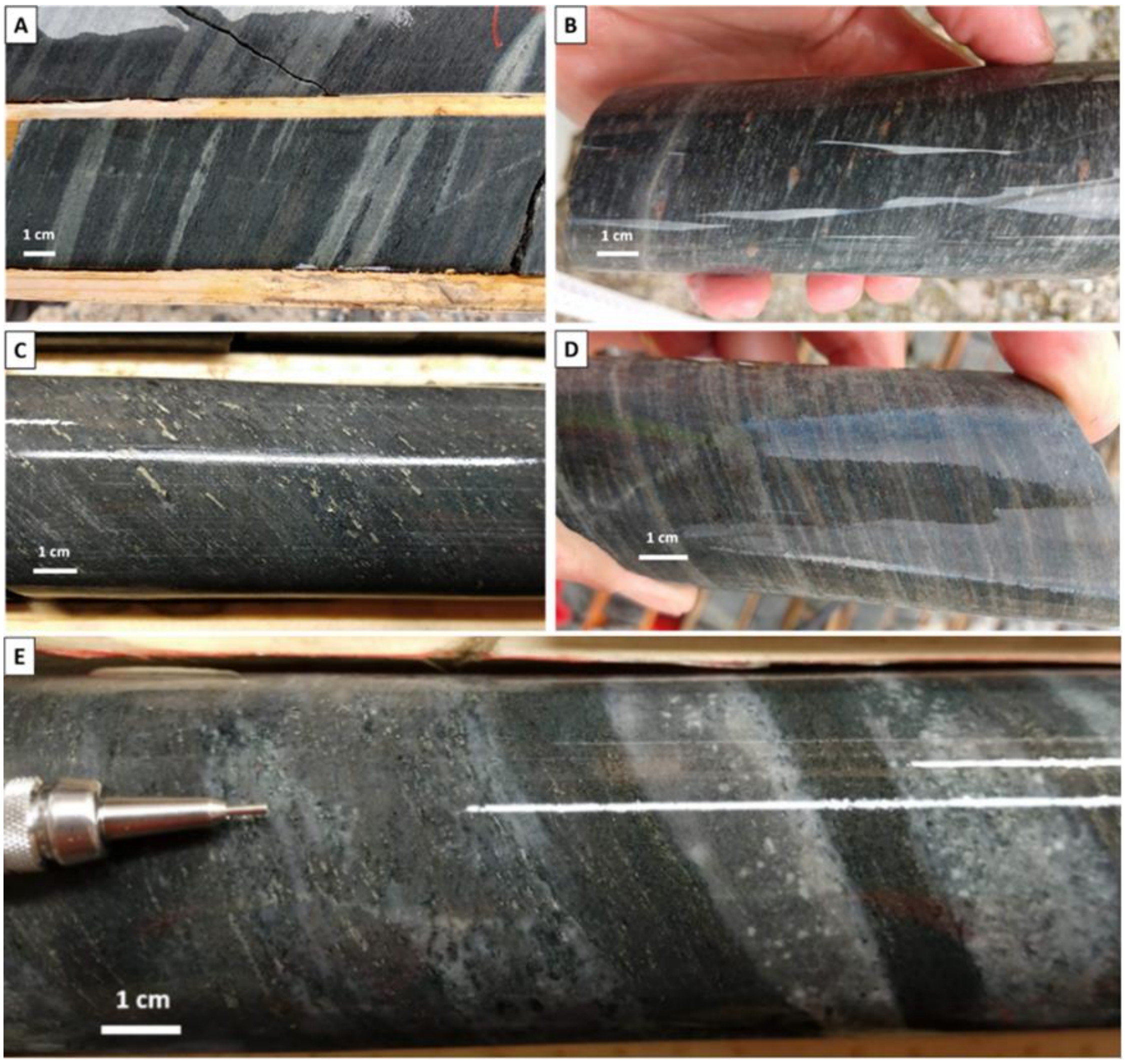
<u>87 and J Zones - Mafic to Intermediate Volcanic Sequence</u>

Dominantly occurring throughout the entire Property, and surrounding the Troilus deposit region, is a thick sequence of volcanic rocks of variable composition (Figure 7-6). The lower footwall region is dominated by mafic volcanics, essentially represented by massive and/or pillow basalts. The primary volcanic textures are rarely identified, being completely transposed by a strong regional foliation. This tholeiitic basalt sequence is overlain by a comparatively thin layer of transitional basalt. This unit is commonly pillowed and can also be seen intercalated within the overlying intermediate volcanic package. These relationships demonstrate the gradational transition from tholeiitic/transitional mafic volcanics to calc-alkaline intermediate volcanics.

The intermediate volcanics present as a sequence of banded / laminated and porphyritic to thick medium-grained flows with intercalated volcaniclastic, tuffaceous and volcanic breccia horizons. The banded / laminated volcanics display quartz-feldspar-rich bands and layers that are dominant over light-green amphibole layers. Porphyritic flows contain both feldspar and amphibole phenocrysts, the latter of which can be multiple centimetres in length and resemble lapilli when deformed. The J zone is host to a series of thicker, more medium-grained flows, which can resemble a finer-grained diorite. Garnet and quartz-rich intervals of volcaniclastic rocks occur toward the top of the sequence, as well as amorphous quartz-bands that could represent exhalative horizons.

The contact between the volcanic sequence and the diorite intrusion in the Z87 and J zones region is difficult to identify and appears to be gradational, with fine to very fine grained and laminated rocks, affected and transposed by intense deformation and hydrothermal alteration.

Figure 7-6: Z87 Zone and J Zone Drill Core Photographs; showing mafic to intermediate volcanic sequence



Source: Troilus (2019)

A. Mafic to intermediate volcanics; footwall of Z87 South

B. Volcaniclastic rocks, quartz-feldspar-garnet rich; hanging wall of J zones

C. Laminated intermediate volcanic rock, mineralized, hanging wall of J zones (J5 sequence)

D. Intermediate, laminated volcanics, Allonge Zone (northern continuity of J zones)

E. Volcanic breccia with porphyritic clasts and amphibole-rich matrixUPDATED)

Diorite and Brecciated Diorite

The dioritic unit forms an elongated body oriented in the northeast-southwest direction with a six-kilometre strike length and a one-kilometre width, surrounded by the volcanic sequence. It comprises a pale to greenish-grey rock, composed predominantly of medium to coarse grained crystals of plagioclase and hornblende.

The Z87 hanging wall transitions from massive to fractured to brecciated diorite, which has been locally observed in drill core, as well as boulders and outcrops around the historic open pits (Figure 7-7). Breccia fragments vary in size from less than one centimetre to over ten centimetres in diameter, are commonly rounded, and are usually elongated parallel to the main foliation and lineation. In less-deformed portions, the fragments are mostly subangular in shape. The matrix is amphibolitic, being primarily composed of fine-grained amphibole and biotite, and minor epidote, quartz, and feldspar grains.

The mapped surface contact between the metadioritic pluton and the surrounding volcanic sequence is projected from drill cores, and it is described as a gradational contact. The outer margins of the metadiorite grade into the fine-grained intermediate to mafic laminated rock.

The plutonic nature of this unit was first postulated by Carles (2000), which stated that “well-developed igneous textures” (coarse grained phases) and the absence of extrusive features would suggest a plutonic nature, possibly emplaced at shallow depth. The fine-grained diorite could also locally be the result of grain size reduction during deformation. An analysis of the litho-geochemistry dataset available for the Troilus deposit (Carles, 2000; Larouche 2005) shows several distinct compositions among diorite samples that are associated with the observations of variable textures. These observations strongly suggest a polyphase intrusive history for the Troilus Dioritic suite, yet a more comprehensive and detailed study is required (Diniz, 2019).

U-Pb zircon dating for the diorite yielded an age of 2791 Ma±1.6 Ma (D. Davis, pers. Commun. In Goodman et al., 2005), making it the oldest age-dated rock unit in the Troilus region.

Figure 7-7: Z87 Zone Photographs; showing diorite and brecciated diorite

Source: Troilus (2019)

A. Brecciated diorite: block on the waste pile located north of the Z87 pit. Note the elongated aspect ratio of the dioritic fragments, parallel to the penetrative foliation.

B. Typical coarse-grained diorite

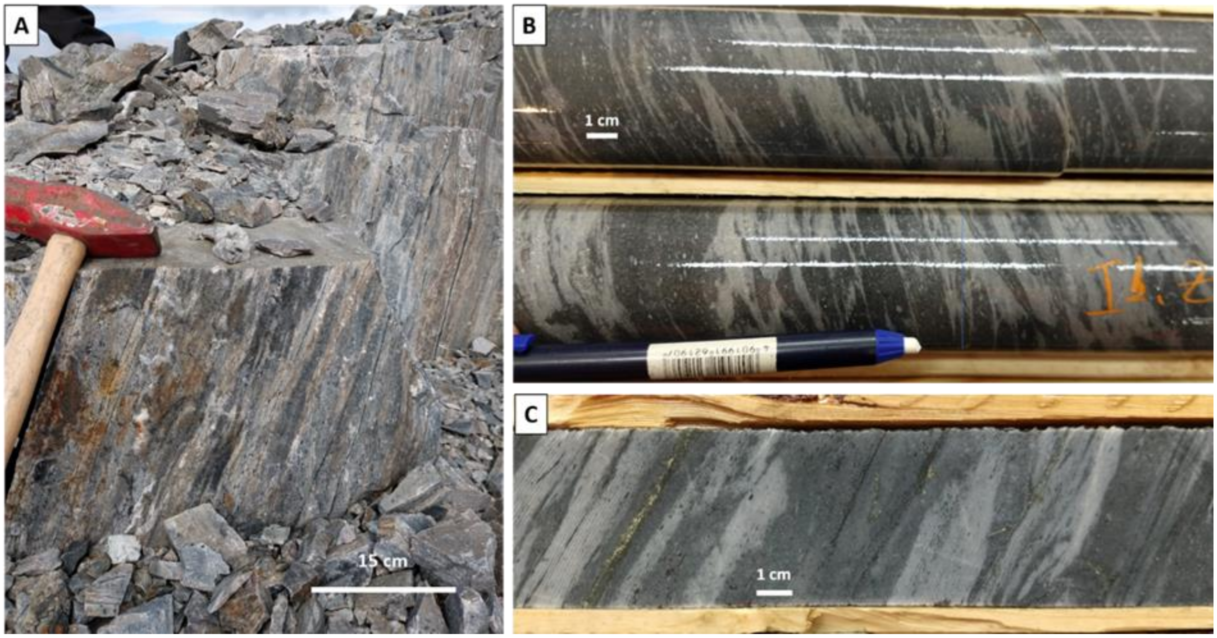
C. Magmatic breccia

Felsic Dykes

Felsic dykes crosscut the volcanic sequence, diorite, and brecciated diorite, with sharp contacts transposed parallel to the foliation. They occur predominantly around the margins of the dioritic intrusion, consisting of several discontinuous bodies, elongated parallel to subparallel to the main foliation. The felsic dykes are typically porphyritic with feldspar and lesser quartz phenocrysts that are commonly destroyed when the rock is highly sericitized and sheared (Figure 7-8).

Two main decameter-thick felsic dykes occur at Z87, comprising the footwall and hanging wall of the main mineralized zone. In the J zone, the felsic dykes occur mainly in the immediate hanging wall, are discontinuous, and occur in an anastomosing pattern, up to ten metres thick. [In the southwest end of the Z87 Zone] is dominated by felsic dykes, up to several metres thick, occurring in an anastomosing and locally stockwork-like pattern.

Figure 7-8: Z87 Zone Photographs; showing felsic dykes



Source: Troilus (2019)

A. Felsic dykes in outcrop, Z87 pit; massive to slightly laminated

B. Porphyritic felsic dyke showing sericite alteration overprint; apparently transposed by the main foliation, Z87 Zone

C. Mineralized massive felsic dyke showing silicification and sericite alteration, Z87 Zone

They are variably affected by biotite alteration and by overprinting muscovite alteration. The latter forms a stockwork, probably corresponding to fracture networks. Increasing muscovite alteration may have reduced the competency of the felsic lithology resulting in it being preferentially deformed. Zones of intense muscovite alteration are strongly foliated, and give a banded texture, which can lead to confusing the dikes with a felsic tuff.

Magmatic zircons in one large felsic dike in the footwall zone of the Z87 zone orebody have been dated and yielded an age of 2782 Ma±6 Ma (Dion et al., 1998 in Goodman et al., 2005; Pilote et al., 1997 in Carles, 2000).

Granitic Intrusions

The Troilus deposit is located in the vicinity of major granitic intrusions: to the east (the Parker pluton) and to the south (the Parker Junior pluton). Pegmatite, granite dikes, and large granite bodies are observed in drill core, and in the Z87 and J open pits. They are present over intervals measuring a few centimetres to over 100 m in thickness. The main granite bodies are observed at depth to the northeast of, and below the Z87 gold trend. They are referred to as the footwall granite.

These intrusive units generally overprint the regional foliation at the sample / core scale. The foliation is observed to be contorted around the granitic bodies at the regional scale. This suggests the granite bodies were emplaced after the formation of the foliation in a late-to post-tectonic timing and their emplacements warped the pre-existing foliation while the rocks remained ductile. A preliminary U/Pb age date of 2698 Ma was determined for titanite from the Parker granite (Goodman et al., 2005).

7.3 Structural Geology

The Troilus deposit is hosted in a zone of intense deformation and experienced upper-greenschist to lower-amphibolite metamorphic conditions. At least two regional phases of deformation are recognized in the Troilus deposit region.

7.3.1 Deformation Phase D1

The main deformation features at Troilus correspond to a west-northwest to east-southeast ductile flattening event referred to here as D1. The main planar structure is a pervasive and ubiquitous foliation, S1. It affects most lithological units at Troilus, except for the post-tectonic granitic bodies. It is oriented N60°E on average, and dips 55° to 70° towards northwest, being slightly steeper in the J zones when compared to the Z87 and Z87S.

Local variations in the foliation orientation could be related to the foliation deforming in proximity to the competent Parker and Parker Junior intrusions. The intensity of the foliation also varies among the different lithologies. Coarse grained diorite is mostly unaffected to weakly foliated. The foliation is stronger in zones of biotite or muscovite alteration, suggesting the deformation is enhanced in altered, auriferous, and less competent zones.

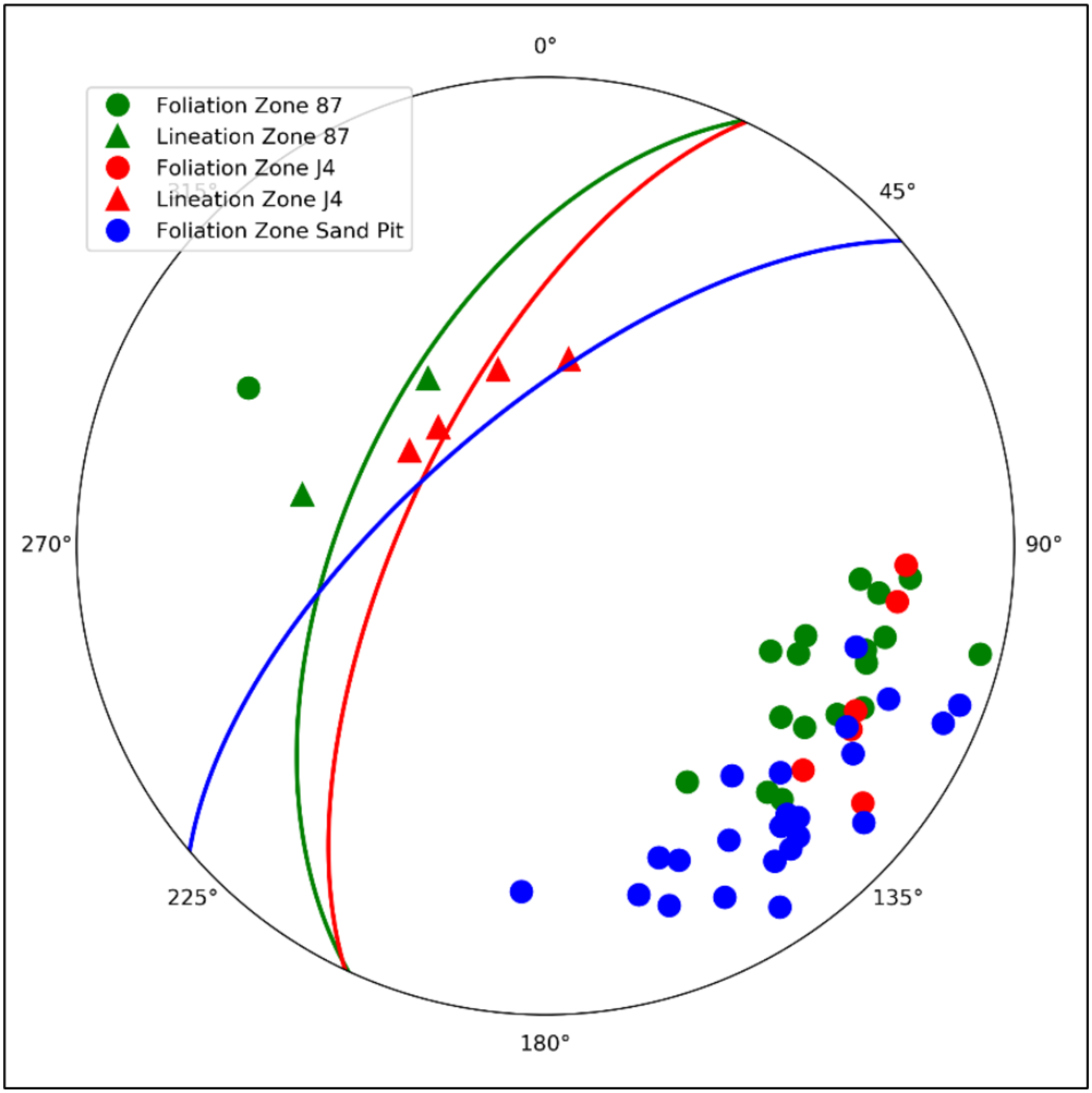
Pre-D1 planar features such as veins, veinlets, and stockworks are variably transposed parallel to the S1 foliation. Similarly, bedding or volcano-sedimentary layering, and geological contacts are transposed parallel to the S1 foliation.

Tight isoclinal F1 folds are associated with an axial planar S1 foliation, and some of these F1 folds can be rootless, illustrating that strong transposition occurred during D1. Fold axes are subparallel to the stretching lineation indicating a strong transposition. This orientation is likely to produce a downdip plunge of gold mineralization parallel to the stretching lineation. The intensity of the deformation and the tight and isoclinal nature of the folds hamper the observation of F1 fold hinges but folding in the Troilus deposit is probably ubiquitous at various scales.

A down-dip stretching lineation oriented -60°/322°Az within the foliation is observed to affect diorite breccia fragments. Biotite and amphibole are preferentially oriented parallel to this lineation. The X:Z stretching ratio from breccia fragments is estimated at 6:1 and the Y:Z flattening ratio is estimated at 3:1, illustrating a strong flattening perpendicular to the foliation combined with a moderate stretching component along the lineation.

Figure 7-9 presents the stereonets of main planar and linear structures at the Project.

Figure 7-9: Stereonet of the Main Planar and Linear Structures – Troilus Gold Project



Source: Troilus (2018)

7.3.2 Deformation Phase D2

At the deposit scale, the second phase of deformation, D2, is marked by northeast-southwest striking, steep-dipping shear zones, identified in the Z87, SW, and Z86S zones. These shear zones are at a low angle with the S1 foliation and crosscut the S1 foliation and quartz veins.

On a regional scale, this second deformation phase also corresponds to important deformation corridors with an interpreted dextral sense movement, La Fourche and Dionne fault zones (Simard, 1987; Gosselin, 1993; Gosselin, 1996), which locally cut and segmented the Troilus Syncline (F1 fold). The zones are characterized by local centimetric to metric isoclinal folds that affect the main regional schistosity, forming a crenulation cleavage. Locally a pronounced sub-horizontal stretching lineation can be observed. The Parker fault zones may also have been formed during D2 and represent a complex array of inverse faults, oriented mainly parallel to bedding and to the main regional foliation, occurring in the north-northwest border of the region, marking the contact zone with the granite-gneiss terrane. A high angle stretching lineation verging to the southeast is normally observed (Gosselin, 1993).

7.3.3 Late NNE-SSW Brittle Faults

A series of sulphide-bearing brittle faults are present on the north wall of the Z87 pit. These faults are thin fault zones (less than 0.5 m in width) characterized by a strong muscovite alteration, silicification, and the presence of sulphides. These faults are oriented subparallel to the foliation and are regularly spaced in the pit, with one every 20 m to 50 m. They are commonly present at the contact between felsic dykes and the breccia. Down-dip slickensides, reverse displacement of pegmatite dykes, and sub-horizontal to moderate northwest dipping quartz tension veins all indicate a reverse movement. The presence of muscovite, quartz, and sulphides suggests that these are sericitic faults zones that were interpreted as hosting part of the gold mineralization at Troilus, as described in Goodman et al. (2005). No significant increase in gold grade was associated with these fault zones in drill core however, suggesting they are not a significant host of the gold at Troilus. Their brittle nature, and the crosscutting relationship with pegmatite dykes indicate these faults are probably part of a possible younger D3 deformation phase.

7.3.4 Fractures

Three main fracture orientations are mapped in the deposit area (SRK, 2018). The first set, oriented at azimuth 025° and dipping at -65° west, is subparallel to the regional foliation and represents the major fracture system in the Z87 pit area. The other two sets (035°/25° and 320°/85°) cut the regional foliation almost at a right angle. The combined effect of these fractures has induced local instability in the Z87 pit. Faulting is observed locally in the pit. The main orientations of the faults are 240°/-55° and 160°/-60°. These two fault orientations do not cause any overall wall stability concerns but may create problems locally.

7.4 Mineralization

The main mineralized zones at the Property occur around the margins of the Troilus Diorite, and comprise the Z87 Zone, the J Zone, and the SW Zone. Other important mineralized zones discovered to date include the northern continuity of the J Zone, named the Allongé Zone, and the southwestern margin of the metadiorite.

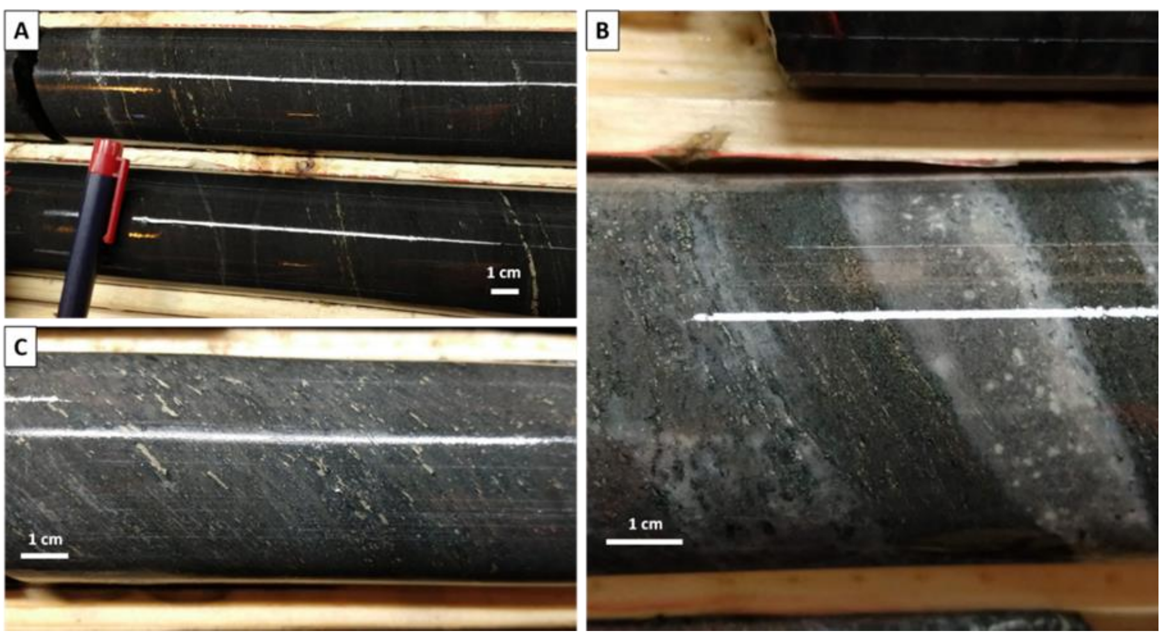
Troilus is primarily an Au-Cu deposit, but contains minor amounts of Ag, Zn and Pb, as well as traces of Bi, Te, and Mo. Gold-copper mineralization at the Troilus deposit comprises two distinct styles, disseminated and vein-hosted. Gold mineralization is spatially correlated with the presence of sulphides, even though the sulphide content does not directly correlate with gold and copper grade. The main host rocks of the disseminated mineralization are the mafic to intermediate flows and tuffs. Vein-hosted gold-copper mineralization is found within the volcanic rocks as well as the felsic dykes and diorite pluton.

7.4.1 Type 1 – Disseminated Mineralization

Disseminated mineralization (Figure 7-10) comprises most the deposit's gold and copper content (>90%, Goodman et al., 2005). Gold and copper are predominantly associated with fine grained disseminated sulfides and/or millimetre wide sulfide streaks and stringers parallel to the main foliation, comprising between 1 % by weight and 5% by weight of the rock. The most abundant sulfides are pyrite, chalcopyrite, and pyrrhotite, with minor amounts of sphalerite, predominantly in the SW Zone. There are trace amounts of bornite, galena, and arsenopyrite locally.

The gold generally occurs as electrum, containing up to 15% by weight silver (Goodman et al., 2005). It is found between sulphide grain boundaries, usually chalcopyrite and pyrrhotite. Petrographic studies undertaken by M.Sc. students at the University of Western Ontario revealed a simplified mineral paragenesis of euhedral pyrite followed by subhedral pyrrhotite, electrum and possible gold-silver telluride and bismuth compounds, and lastly anhedral chalcopyrite and bornite mineralization. This mineralization style is found across each of the zones, and mainly occurs pervasively throughout the mafic to intermediate volcanic rocks, but it is also observed in the brecciated margins of the diorite and, more recently (2022), within and around shear zones in massive diorite. It is generally associated with biotitic-chloritic alteration. There are metric-scale intervals of strong sericitization with greater amounts of coarser grained pyrite, but these do not appear to be related to gold or copper mineralization.

Figure 7-10: Z87 Zone and J Zone Photographs; showing disseminated mineralization



Source: Troilus (2019)

A. Disseminated pyrite in a fine grained, biotite-rich intermediate volcanic rock- J Zone

B. Intermediate volcanic breccia; fine sulfides disseminations in the amphibole-biotite-rich matrix - Z87 Zone

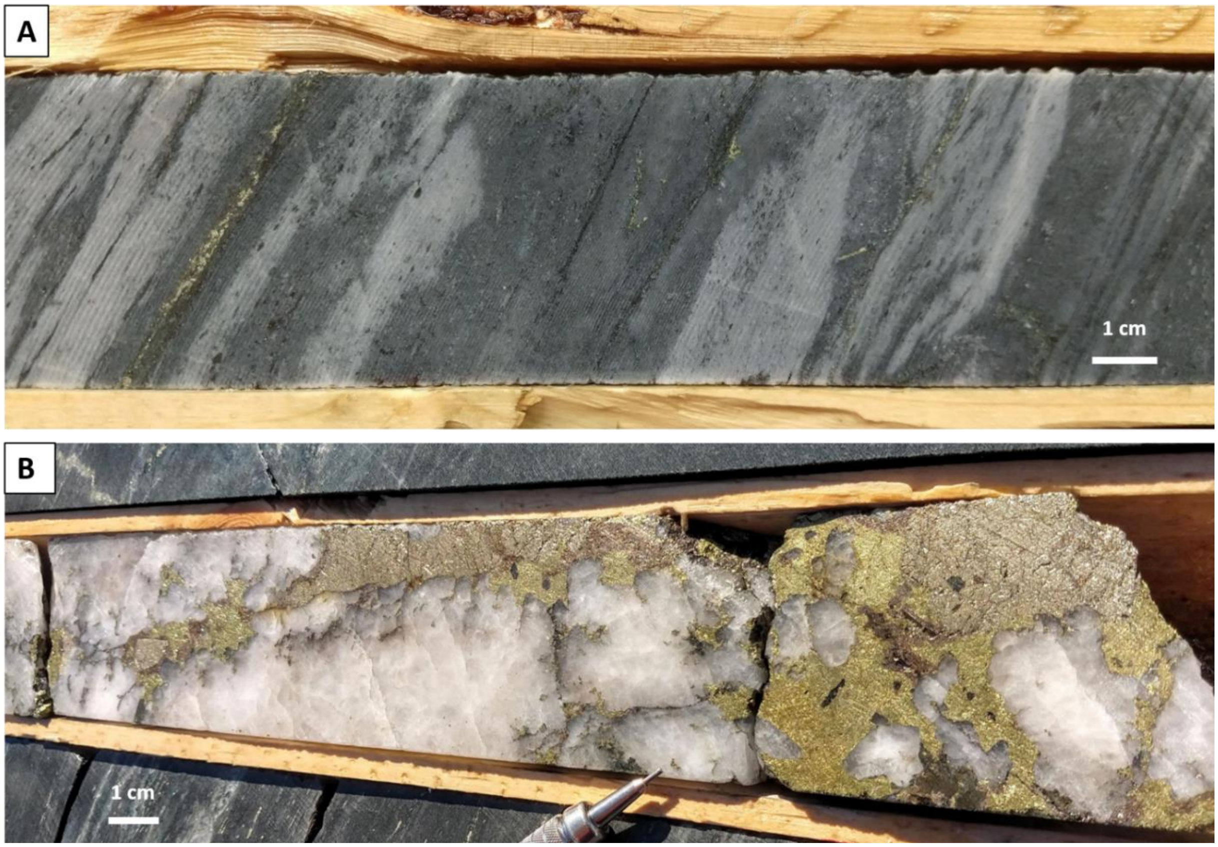
C. Disseminated medium grained pyrite in volcanic laminated rock - J5 zone J Zone

7.4.2 Type 2 – Vein-hosted Mineralization

This mineralization style is characterized by gold bearing veins, with gold mineralization restricted to the veins and veinlets, and is classified as gold-only, since copper mineralization is rare and erratic (Carles, 2000). This type of mineralization is reported to be hosted in all rock types occurring within the mineralized envelope in the Troilus deposit (Figure 7-11).

Several generations of gold-bearing veins have been identified and described by Goodman et al. (2005), and Larouche (2005), the latter especially focused on J zone. With regards to grade and abundance, the most significant are quartz-chlorite (±tourmaline) veins. These veins occur in silicified wall rocks to sericitized high strain zones which cut the main foliation and the margins of felsic dikes, and within 1 m to 15 m wide shear zones in the diorite pluton. Gold-bearing millimetre-to centimetre wide veinlets are locally present as swarms parallel or subparallel to spaced cleavage in the silicified rocks. The veinlets contain free gold and minor amounts of sulphide. Much of the gold is fine grained and contains up to 20% Ag, however, gold grains can be up to greater than 1,000 µm in size. Gold is also observed within fractures on the boundaries of coarse euhedral pyrite grains. Locally, a second set of gold bearing quartz veinlets cut the first. These carry fine grained gold (greater than 95%) and minor pyrite, chalcopyrite, sphalerite, galena, and Te- and Bi-bearing minerals, including tellurobismuthite (Bi2Te3), calaverite (AuTe2), and hessite (Ag2Te). Although volumetrically much less significant than the main disseminated mineralization, the veinlets can contain grades greater than 50 g/t Au over a one metre interval. Coarse grained gold recovered by a gravity circuit in the mill accounted for about 30% of the gold produced. Presumably much of this coarse gold was derived from the veins. High grade shoots related to the veinlet zones are oriented 40° clockwise from the main disseminated mineralization.

Figure 7-11: Z87 Zone and J Zone Photographs; showing vein-hosted mineralization



Source: Troilus (2019)

A. Millimetric Py-Po-rich veinlet in an altered felsic dike (sericitization and silicification) - Z87 Zone

B. Atypical very high-grade quartz veins, up to over 1-m thick; remobilized pyrite-chalcopyrite-pyrrhotite - J Zone

7.4.3 Alteration

Gold mineralization at Troilus is associated with various types of alteration described below.

Biotite

An early, pervasive, weak to strong biotite alteration affects the diorite, breccia, and felsic dykes. The matrix of the breccia is preferentially altered. This alteration style is widespread in the deposit and can extend up to tens of metres away from the main gold zones. Sulphide content in drill core increases with biotite alteration intensity, suggesting a genetic link between the two. The biotite is transposed parallel to the foliation, indicating alteration occurred prior or during the main deformation event. The foliation intensity increases in strongly biotite altered intervals, due to the lower competency of the biotite-bearing rocks.

Muscovite

The vein-hosted mineralization is spatially related to a strong sericitization within the high strain zones, better developed in the felsic dikes, reaching up to several centimetres (Carles, 2000). Sericitization is also present in the amphibolite and the matrix of the breccia. A weak to strong muscovite alteration is present in some felsic dykes and varies in texture from pervasive to stockwork. It also locally alters the diorite and the breccia. Gold mineralization can be present in muscovite altered rocks, but sulphide content does not increase with the presence of muscovite alteration. Muscovite stockwork-like textures are locally transposed by the main foliation, indicating muscovite alteration occurred after biotite alteration but prior or during the main deformation event. Zones of higher foliation intensity, and thus of higher deformation, occur in strongly muscovite-altered rocks, probably due to the lower competency of these lithologies compared to unaltered rocks. The most highly deformed and sericitized parts of the rock are commonly surrounded by a silicified envelope that could reach several metres in width.

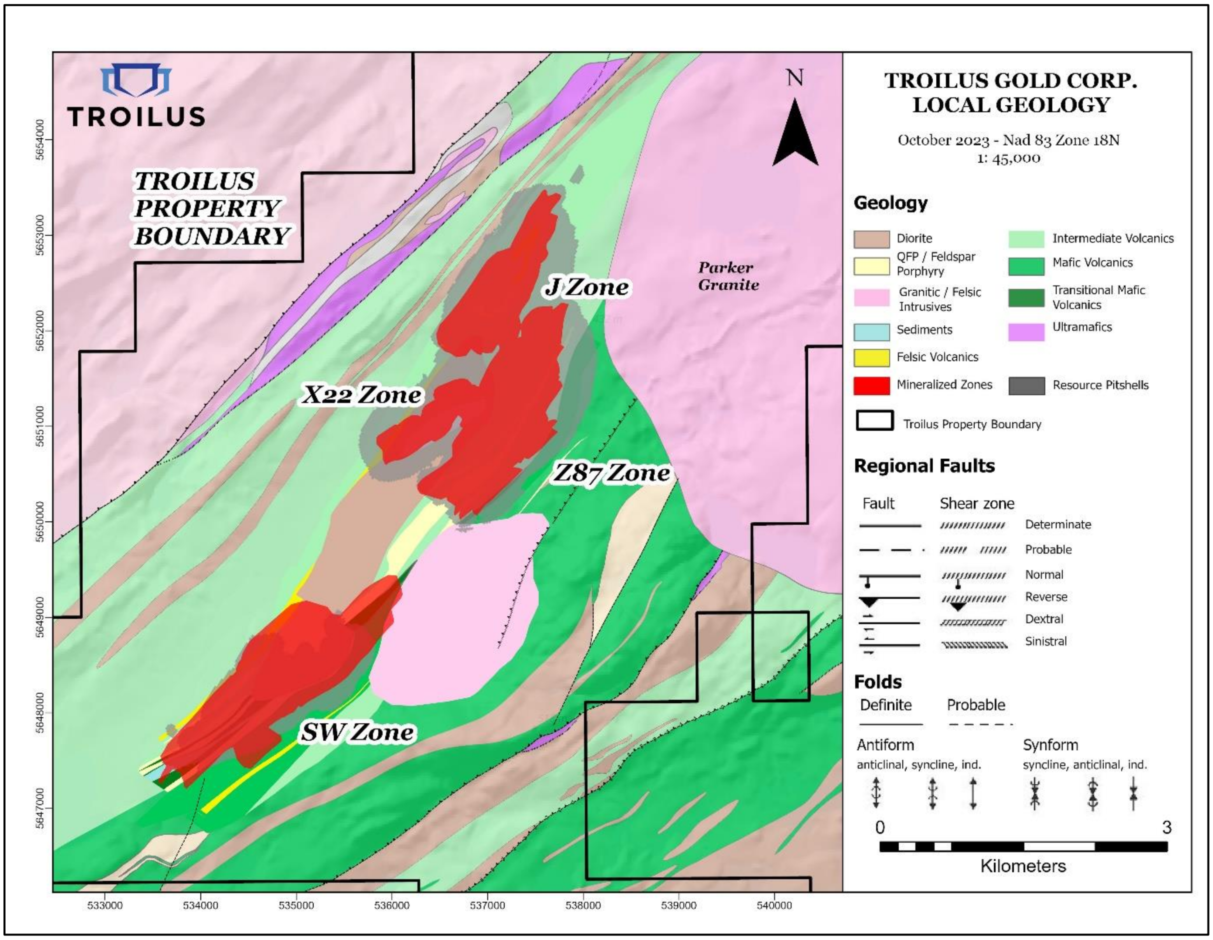
Calcic Metasomatism

A syn-deformation epidote-amphibole alteration occurs both pervasively and as veins in the deposit area. It consists of pervasive calcium-rich minerals such as calcium amphiboles, epidote, or calcite occurring in two metre- to ten metre intervals in drill core, or in discrete layers or bands measuring less than 20 cm. Veins of quartz, calcite, epidote, grossular garnet, and diopside may also be locally present. Gold mineralization is present locally in calc-silicate altered rocks, however, barren calc-silicate altered rocks also occur. Calc-silicate bands and veins can be parallel to the foliation, folded by the main deformation event, or can crosscut the foliation, all indicating that calc-silicate alteration occurred during the main deformation event.

7.5 Mineralized Zones

The four principal deposits of the Project are the: Z87, J, X22 and SW Zones. Figure 7-12 presents the location map of these deposits

Figure 7-12: Principal Mineralized Zones – Troilus Gold Project



Source: Troilus (2023)

7.5.1 Z87 Zone

The main pit of the Troilus Mine, operated by Inmet from 1996 to 2010, was developed in the Z87 orebody. The mineralization in the Z87 occurs as a series of anastomosing lenses, extending for approximately 1,300 m along strike from 12,900N to 14,200N with variable thickness and locally reaching over 100 m wide. With increasing depth, individual mineralized lenses coalesce to form a single sheet-like body that was approximately 40 m thick on average (Fraser, 1993).

The long axis in the Z87 is oriented N35°E with the orebody dipping to 55° to 65° northwest, from southwest- to northeastern portions, respectively. Detailed studies of Z87 blasthole data and diamond drill intersections revealed the presence of higher-grade shoots, which plunge to the west-northwest at ~55°. This primary plunge is controlled by the D1 stretching lineation. A secondary enriched trend has been identified and corresponds to the intersection of D2 shearing and the S1 fabric, which plunges at ~30° to the southwest.

In Z87, the peak of enrichment in gold and copper overlap but are not exactly coincident. A metal zonation is observed, associated with the sulfide content. The structural footwall is enriched in a chalcopyrite-pyrrhotite assemblage, with copper more abundant than gold. This zone grades into an intermediate pyrite-chalcopyrite zone, which comprises the main ore zone of the deposit and contains gold and copper. The structural hanging wall is dominated by pyrite and is gold-rich relative to copper. The variable gold and copper relationship may also be due to the cross-cutting nature of D2 structures, which are believed to control "stage 2" mineralization. The sulfide assemblage may also reflect a zonation in temperature between a higher temperature footwall zone dominated by chalcopyrite and pyrrhotite that transitioned towards a lower temperature pyrite dominant hanging-wall zone. The origin of this zonation is unknown but could either be primary and directly linked to the genesis of the deposit, or secondary when the deposit was metamorphosed during the regional deformation event, or possibly by the heat induced during emplacement of the nearby granitic plutons.

7.5.2 J Zone

The J Zone orebody hosts two parallel mineral zones formerly known as J4 zone and J5 zone. J4 is the smaller of the two formerly mined open pits along with the main Z87 zone. Like other zones at Troilus, mineralization in the J Zone is associated with feldspar porphyry dykes, as they help to focus strain and fluid flow during mineralizing processes. Unique to the J zone is the impact of more massive flows and diorite dykes within the intermediate volcanic package, which help in the development of structural traps, similar to the feldspar porphyry dykes.

Locally, feldspar porphyry dykes can produce more pronounced fold patterns than is typical of rocks at Troilus. The hinges of these folds contain some of the highest grades in the J Zone.

The main mineralized intervals in the J Zone are characterized by sulfide stringers and fine sulfide disseminations along the foliation occurring within very fine-grained, biotite-rich, and silicified intermediate volcanic rocks. Pyrite is the main sulfide, and it is intrinsically associated with gold mineralization.

Compared to Z87 Zone, the J Zone has a lower copper grade, more free-gold, and dips more steeply at ~65° to the northwest. Higher-grade ore "shoots" are parallel to the stretching lineation. Mineralized trends are observed parallel to the S1 fabrics, as well as in cross-cutting shear zones. At least one of these cross-cutting structures extends into the Z87 pit.

7.5.3 X22 Zone

The X22 Zone is situated adjacent to the west of the Z87 Zone and approximately 200 m southwest of the J Zone. The X22 Zone is the only economic zone hosted entirely within the Troilus intrusion. Mineralization is dominantly constrained to a more felsic (tonalitic) section of the intrusion that lies along a D2 structural corridor. Zones of Au-Cu enrichment are located at intersection with D1 structures and are associated with broader biotite and silica alteration, with more discrete albite alteration and shear hosted sericite alteration. Sulfides are both disseminated and vein-hosted, and are primarily pyrite, pyrrhotite and chalcopyrite, with lesser sphalerite, galena and molybdenite. A network of shear and extensional veins containing variable amounts of quartz, carbonate, sulfide minerals, biotite and tourmaline is also present in the mineralized zones. The tonalite often displays blue quartz phenocrysts when highly strained and altered.

Mineralization is most well-developed near the footwall contact of the tonalitic body. Locally, felsic porphyry dykes are host to high grades (>10 g/t Au). The northern-most mineralization in X22 is Cu-poor relative to the rest of the zone and is associated with these felsic dykes. The southern extents of X22 Zone is host to relatively abundant massive sulfide veins containing grades in excess of 100 g/t Au. The southern "tip" of the stretched ellipsoidal tonaltitic body is an affective low-pressure trap for mineralizing fluids, as it contains the highest grades in X22 Zone. Mineralization dips at ~55° to the west, with higher grade "shoots" oriented parallel to the D1 stretching lineation. The tonalite itself is also stretched parallel to this lineation.

7.5.4 Southwest Zone (SW Zone)

The SW Zone is situated approximately 3 km southwest of the Z87 Zone.

As observed in all main mineralized zones on the Property, the SW Zone lithological sequence is comprised by a dominantly mafic volcanic footwall, and a more intermediate to felsic hanging wall. This volcanic package is intruded by syn-volcanic dioritic and felsic rocks. Disseminated gold-copper mineralization is found within the upper portion of the mafic footwall rocks and the overlying intermediate volcanic hanging wall. Vein-hosted gold mineralization, as previously described, is found principally within and around the felsic dykes and diorite.

The footwall mafic volcanic sequence in the SW zone represents a homogeneous package, composed of dark green, amphibole-rich, fine-to locally coarse-grained rocks. Locally, it contains sericite and sulfide-rich metric to decametric intervals, laminated/banded, occurring mainly within the upper part of the sequence. These intervals are normally anomalous in Au, Zn, Ag, S. The dominant sulfide is pyrrhotite.

Intrusive felsic rocks occurring in the SW Zone comprise mainly two different lithotypes: (i) rhyolitic felsic dykes ("Feldspar Porphyry" or "Felsic Dykes") and (ii) younger dacitic felsic dykes ("Intermediate Feldspar Porphyry"). They share similar compositional and textural characteristics and are often mistaken due to the lithological similarities and alteration pattern. Both the felsic dykes and intermediate feldspar porphyry units show porphyritic textures, with felspar phenocrysts dispersed in a quartz-rich groundmass. Intense silica and sericite alteration are commonly observed in both units.

Felsic dykes are thinner and occur as "arrays" of several dykes, crosscutting the sequence, and often concentrated in the contact zone between mafic footwall and more intermediate to felsic hanging wall.

The intermediate feldspar porphyry defines a continuous unit, tens of meters thick, occurring immediately above the mafic footwall sequence. It hosts an important part of the mineralization found in the eastern domain of the SW Zone. It is generally lower grade, and relatively copper-poor compared to the mineralized intervals observed in the magnetite-rich breccia occurring in the hanging wall and footwall of the unit.

A magnetite-rich and highly silicified transitional basalt (the "Southwest Breccia") represents the main host rock for gold and copper mineralization at the SW Zone. The unit varies texturally from medium-sized pillows with thin selvages, intervals of subangular lapilli tuff, and containing 1 to 20 cm subangular xenoliths of both epidote and a porphyritic felsic rock. The presence of these fragments locally and the volcanic textures exhibited are informally referred to with the structural term, 'breccia.'

Sulfides and quartz are often filling fracturing and locally forming stockwork-like textures within the magnetite-rich silicified fragments. High-grade zones are copper-rich and reach up to 10 m to 20 m thick.

Intermediate Feldspar Porphyry dykes occur intercalated with the brecciated, sulfide and magnetite-rich intervals.

The SW Zone is defined by two key mineralized zones: the 'Main Zone' and the 'West Zone.' The Main and West Zone are predominantly differentiated by gold content and have been interpreted to represent opposite limbs of a major regional syncline that has likely been subjected to a primary, regionally emplaced phase of gold bearing mineralization (first major gold event). The Main Zone distinguishes itself from the West Zone having clearly been highly altered by a secondary / later gold and copper bearing event, which is characterized by dark silica (quartz) flooding, brecciated (fractured) fragments, and intense fracture-filling chalcopyrite (main source of copper) and pyrite, pervasive magnetite, as well as free gold.

Higher grade intervals appear associated with the highly altered Main Zone resulting from local, focused structural controls and fluid traps acting as a conduit for alteration / mineral deposition.

The SW Zone and Z87 show important similarities in terms of host rocks, mineralization style and geochemistry, as summarized below:

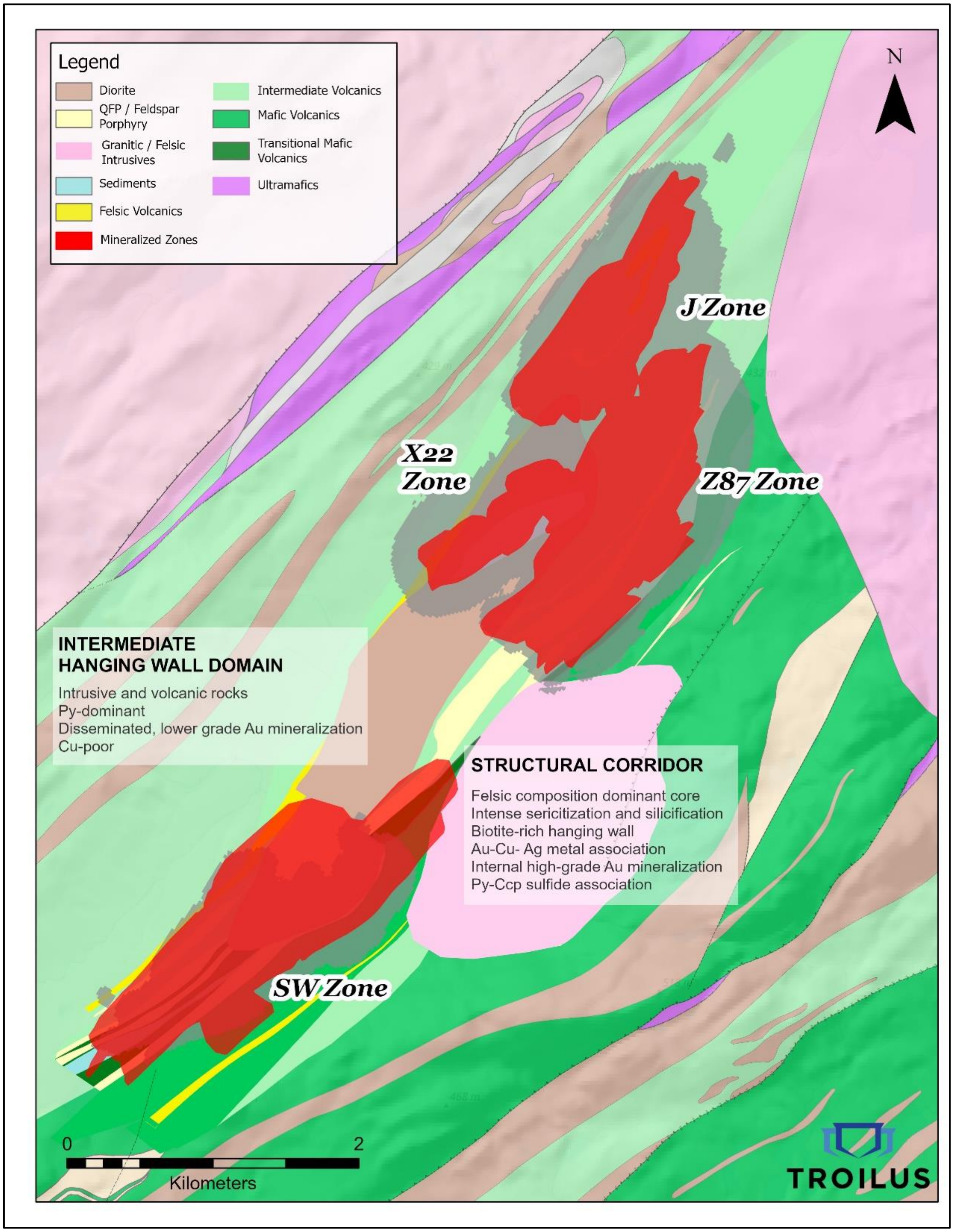
- similar Au-Cu-Ag metal association
- high-grade Au associated with chalcopyrite (filling micro-fracturing and in sulfide margins)
- zoning: pyrite-rich hanging wall, Pyrite-Chalcopyrite core zone, Pyrite-Pyrrhotite foot wall
- similar host rocks:
  - brecciated intermediate volcanic hanging wall higher grade, Au-Cu association
  - cross-cutting felsic dykes
  - least altered, medium to coarse-grained diorite in the hanging wall, with strongly silicified and sericitized shear zones with vein-hosted gold
  - amphibolite-grade mafic volcanic foot wall

Both zones are located within the same structural corridor represented by the eastern limb of the interpreted Troilus syncline, comprising an intensely altered and deformed sequence, with a dominantly felsic "core," separating two distinct domains: a mafic-dominant footwall, and the intermediate volcanic/intrusive package in the hanging wall.

The similarities between the two zones reinforce the potential to expand mineralization towards the underexplored 3.5 km linear trend that separate Z87 and SW Zone, the "Gap Zone".

Figure 7-13 presents a schematic and simplified representation of the different domains hosting mineralization on the Property. It also highlights a structural corridor that links the Z87 Zone and SW Zone marked by similar mineralization style, host rocks and geochemical signature.

Figure 7-13: Simplified Geology Map of the Mineralized Zones, highlighting the structural corridor



Source: Troilus (2023)

7.6 Prospects/Exploration Targets

7.6.1 Allongé and Carcajou

The Allongé target is situated adjacent to the northeast, along strike, of the J Zone; and the Carcajou target is situated 5 km northeast of Allongé.

The geology of Allongé and Carcajou is typical of the north-eastern portion of the J Zone, it consists predominantly of laminated and strongly foliated intermediate volcaniclastic rocks with subordinate intermediate porphyritic flows, metre to decametre-scale quartz-feldspar porphyry dykes (QFP), and undeformed granitic-tonalitic dykes of the Parker pluton in the footwall. The mineralization style is the same as in the J Zone; gold is associated with pyrite disseminations and stringers concordant with the foliation throughout the volcanic package. Gold mineralization is also found within QFPs and within thin shear zones. Mineralization is not associated strongly with any alteration minerals, though the host rocks are pervasively weakly silicified and sericitized. Very tight, localized folds occur throughout the volcanic sequence, generally with quartz veins along the axial planes. These folds occur throughout the deposit, particularly in the Southwest Zone. Amphibolite-grade metamorphism increases in intensity towards the southwest of the zone, and there are locally thin intervals with pre- to syn-deformational rounded amphibole porphyroblasts.

7.6.2 Beyan and Cressida

The Beyan Target and Cressida Target are situated approximately 7 km and 10 km, respectively, on strike southwest of the SW Zone.

Beyan Zone is characterized from SE to NW: an amphibolitized bimodal volcanic sequence, a mafic intrusive complex, an intermediate volcaniclastic sequence (like the one observed in the J4 Zone) and finally a mafic volcanic sequence.

The bimodal volcanic sequence also contains sedimentary layers and small stratabound exhalative horizons including chert and iron formation (oxide and sulphide variety). The volcanic rocks are intruded by mafic, dioritic, and locally ultramafic sills and felsic dykes (QFP and FP). Basalt commonly displays pillowed and variolitic textures with the inter-pillow material being replaced by silica and carbonates. Rhyolitic flows are characterized by aphanitic to porphyritic textures (quartz eyes), and when sheared they develop a strong schistosity marked by sericite. The volcanic sequence shares some similarities with the mine sequence, including the intermediate banded ash tuff that can be traced to the J4 Zone. The main difference is the absence of a major diorite intrusion (Troilus diorite). Some diorite dykes are observed at Beyan, but they are usually limited in width.

The intermediate banded ash tuff hosts the main mineralization zone at Beyan, it exists in a stratabound and stratiform lens with disseminated to semi-massive sulphides (pyrrhotite, pyrite with minor sphalerite, chalcopyrite and arsenopyrite). Alteration assemblages include a proximal strong silica-sericite ± fuschite, carbonate and tourmaline alteration zone sometimes with aluminosilicate porphyroblasts, garnet-biotite-amphibole ± chlorite and carbonate alterations. The silica-sericite zone is usually spatially associated to quartz-carbonate veins. The mineralized zone appears to follow the regional foliation S1 and stratigraphy (NE-SW). The mineralized horizon corresponds to a high chargeability and low resistivity zone located in between two more competent mafic units of basalt and gabbro. This difference in competency, porosity and potentially chemical reactivity more likely played an important role in focusing the gold-bearing hydrothermal fluids.

Other gold-bearing mineralization corresponds to small sedimentary intervals with arsenopyrite grains in the bimodal volcanic sequence, sulphide-rich iron formations, cherty horizons, and smoky quartz-carbonate veins in mafic units (locally with visible gold). When parallel to the main foliation the quartz veins display evidence of boudinage, while crosscutting veins display evidence of folding. Arsenopyrite appears to be a good indicator for gold mineralization to the SE of the intermediate banded ash tuff.

Cressida lies along strike from Beyan, and the geology is very similar. The primary host of gold mineralization is an intermediate banded ash tuff as observed in Beyan, averaging 40 – 60 m thick and steeply dipping to the northwest. It is bounded to the northwest by a thick package of basalt in gradational contact with the unit. Also in gradational contact to the southeast is the bimodal volcanic sequence as described in Beyan, but predominantly consisting of amphibolitized massive basalt flows with thin felsic horizons. A thinly bedded biotite rich and graphitic sedimentary unit ranging from 1 m to 10 m thick is responsible for the second thin highly conductive IP anomaly to the southeast of the main ore zone. This unit contains up to 30% pyrrhotite and pyrite locally, and strongly resembles the unit observed in the southwestern margins of the Southwest Zone. Interbedded within this unit are the volcaniclastic unit, and 1 m to 2 m thick cherty intervals with significant amounts of grunerite, resembling a silicate facies iron formation. This unit is in gradational contact with amphibolitized basalt to the southeast with significant amounts of carbonate veining. There are some thin unaltered and undeformed mafic-intermediate porphyritic sills intruding the volcaniclastic unit and basalt, appearing more commonly in the northeast and pinching out to the southwest.

Gold mineralization at Cressida occurs primarily within the volcaniclastic intermediate ash tuff, in a stratabound and stratiform pyritic lens 10 m to 30 m thick consisting of disseminated bands of fine-medium grained euhedral pyrite. The pyrite content is approximately 1% to 5% within this zone, with 5 m to 10 m intervals reaching up to 25% and rare 20 m to 30 cm massive pyrite lenses. Increasing pyrite content is spatially associated with silica-sericite ± fuchsite-carbonate alteration. Garnet and aluminosilicate porphyroblasts are common, but not necessarily associated with any mineralization. There is trace pyrrhotite locally within the zone as well. Within the bimodal volcanic unit, there are rare thin intervals of arsenopyrite. The interbedded sedimentary-volcaniclastic-iron formation unit to the southeast contains large amounts of pyrrhotite, and locally minor amounts of pyrite ± arsenopyrite ± sphalerite. Thin concordant and discordant quartz-epidote-carbonate ± calc-silicate and quartz-only veins within the volcaniclastic unit do not appear to be related to gold mineralization.

7.6.3 Testard

The Testard Target is situated approximately 16 km south-southeast of the Project site.

The geology of the Testard Gold Zone consists of east to west of a sequence of intermediate lapilli and block tuff in contact with an ultramafic sill, the contact which represents the regional S0-S1 is oriented NE-SW. Both lithologies are crosscut by a pre- to syn-deformation tonalitic stock (Testard stock). Two generations of shear-zones have been mapped at Testard. The first one corresponds to a major ductile deformation zone linked to D1 and oriented NE-SW that follows the eastern margin of the ultramafic sill. The second generation of shear-zones are more discrete and are oriented E-W, they are interpreted to have formed during the second regional deformation event D2. They consist of brittle to ductile shear-zones that crosscut all lithologies. The contact between the tonalite and the ultramafic sill appear very prospective for gold, silver and locally copper with trace molybdenum and zinc as suggested by the polymetallic nature of the mineralization at the Lac Dauphin showing The previously known gold mineralization at the Testard main showing is hosted within quartz-carbonate-tourmaline veins within a shear-zone at the contact between the tonalite and a mafic-ultramafic dyke oriented northeast to southwest and located 400 m to the east from the main tonalite-ultramafic contact.

7.6.4 Freegold-Bullseye

The Freegold-Bullseye Target is situated approximately 18 km south-southwest of the Project site.

The Freegold Zone is characterized by a complex network of D1 and D2 deformation zones and by numerous precious and base metal showings highlighting the high economic potential of the zone .

The Freegold Zone is a high-grade gold zone controlled by an east to west D2 shear zone which crosscut magnesian basalts of the Crochet member of the Châtillon Formation (Simard, 1981; Gosselin, 1996) to the west and a microgabbro to the east. Proximal to the mineralization the host rock is strongly silicified and sericitized while the distal alteration is chloritic. The altered zones are also characterized by pyrite, chalcopyrite, and small crystals of tourmaline. Field observation in stripped outcrops show that the east-west shear is in fact a sequence of anastomosed and ‘en-relais’ east to west shear zones. The mineralization is contained in quartz veins and massive sulphide veins which contain quartz-pyrite-pyrrhotite-chalcopyrite ± visible gold, they often display a brecciated texture with clasts of quartz. The mineralized zones are usually centimeter to decimeter wide but can locally reach a meter. Boudinaged quartz veins parallel to the main shear and mineralized veins are also observed, but they seem to be more continuous and more present to the east within the microgabbro. Best values from the channels samples at Freegold Zone include 19.3, 16.45, 10.85, 9,72, 7.8, 5.73, 5.42 and 4.5 g/t Au with 42 samples over 0.1 g/t Au. The samples are gold rich compared to silver with trace amounts of copper, bismuth, and cobalt.

7.6.5 Pallador

The Pallador Target is situated approximately 37 km southwest of the Project site.

The geology of the Pallador area consists mostly of a sequence of mafic, intermediate, and felsic volcanic rocks with small horizons of sedimentary rocks intercalated with mafic and ultramafic sills. The area is also characterized by polyphase intrusive bodies such as the Regnault intrusive complex which hosts gold mineralization, and by felsic intrusions of granodioritic to tonalitic composition. This zone is characterized by major east to west and northwest to southeast shear zones and by a complex structural pattern which highlight the presence of multi-kilometric folds. Field observations and data obtained so far indicate that the area has the potential to host different styles of mineralization including gold-silver, copper-zinc, polymetallic (W-Bi-Mo-Cu-Zn with trace Au and Ag) and rare metals (Li-Ta-Sn-Be-Cs-Rb) hosted in lithium-caesium-tantalum (LCT) pegmatites sampled from boulders.

8 DEPOSIT TYPES

The Troilus deposit is better known as an example of an Archean porphyry-type deposit as interpreted in the pioneering work of Fraser (1993). It is frequently cited as such, for example, Robert and Poulsen, 1997; Poulsen, 2000; Sinclair, 2007; Mercier-Langevin et al., 2012; Katz, 2016.

Other interpretations for its genesis include superimposed structurally controlled "orogenic" gold, proposed by Carles (2000) and Goodman et al., (2005). Table 8-1 presents a summary of the main geological characteristics that supported these two models (Diniz, 2019).

Table 8-1: Summary of Mineral Rights for the Troilus Gold Property

Model	Timing	Host Rocks	Sulphides and Metal Associations	Texture/Style	Alteration	References
Au-Cu Porphyry-type	Single stage pre-deformation, pre-metamorphism	In situ hydrothermal breccia, amphibolite, and felsic dikes	Au-Cu zoning; Cu-rich footwall (ccp+po) Intermediate Main ore zone: Au-Cu (py+ccp); Au-rich hanging wall (py)	Disseminated and stringers along the foliation	Main stage potassic alteration (biotite), zoning outwards to a propylic alteration; and phyllic analogous sericitic alteration	Fraser (1993), Larouche (2005)
Multi-stage syn-deformational	Early, pre-peak metamorphism and late, post-peak metamorphism	Early stage restricted to magmatic breccia and amphibolite, Late-stage veins in the breccia, amphibolite, and felsic dikes	Early-stage Au-Cu (py+ccp+po) Late Au-only mineralization (py mainly, sph-gn locally)	Early disseminated and stringer zones Late Qtz-Chl-Tur veins	Main biotic alteration (early stage) Late stage sericitic alteration and silicification halo around quartz veins	Carles (2000), Goodman et al. (2005) Brassard and Hylands (2019)

*modeified from Diniz (2019)

Note: py – pyrite; ccp – chalcopyrite; po-pyrrhotite; gn – galena; sph – sphalerite; tur - tourmaline

The genetic model proposed by Fraser (1993) is based on similarities between Troilus and typical Phanerozoic porphyry deposits. The author interpreted that the biotite-rich zone that accompanied the bulk of mineralization at Troilus would be analogous to the typical potassic hydrothermal alteration core of porphyry deposits being that biotite, the main indicator mineral for this alteration, also occurs in the felsic dikes. Sericite would be the second most common potassium-rich mineral, largely dominant in the felsic dikes.

In Z87, this zone would be centered in the footwall dike and would grade outwards into a propylitic zone, defined by a gradual decrease in biotite and amphibole content, and increase in albite, epidote, and calcite. The alteration zoning would be asymmetric, being better developed towards the hanging wall. Associated with the asymmetrical alteration, a metal zoning marks a footwall dominated by biotitic alteration, and chalcopyrite-pyrrhotite assemblage, being copper-rich, whereas towards the hanging wall, gold would prevail over copper, and would be associated with potassium decrease and sodium increase, and pyrite would be the main sulfide. The in-situ hydrothermal breccia marked the transition, intermediate zone. In addition to what was proposed by Fraser (1993), Boily (1998) suggested that the observed sericitic-quartz association would represent an equivalent of typical phyllic alteration of a porphyry mineralizing system.

Larouche (2005) supports the magmatic-hydrothermal genetic model for the Troilus deposit, although presenting a slightly different chronology of alteration and copper and gold mineralization events. The felsic dikes would have intruded the amphibolite and diorite, followed by brecciation of the host rocks by hydraulic fracturing, and potassic alteration and gold-copper mineralization development. The potassic zone and the mineralization would have been subsequently superimposed by the propylitic alteration, forming late epidote-calcite-quartz veinlets. A final hydrothermal event would have released fluids via felsic dikes, originating a sericitic alteration, better developed in the felsic dikes, and mainly associated with gold mineralization.

Carles (2000) later supported by Goodman et al., (2005), suggested that the Troilus deposit is the result of two superimposed unrelated and structurally controlled mineralization events. The earliest event would be responsible for the introduction of disseminated Au-Cu mineralization in association with biotitic alteration and would be restricted to the mafic rocks (amphibolite, the matrix of the breccia and biotite-rich zones in the metadiorite), only occurring in the margins of the felsic dikes. In the Z87 the mineralization related to this stage would be restricted to a corridor bounded by the felsic dikes. Carles (2000) suggested that the “early stage” mineralization would represent an amphibolite-metamorphic-grade example of “orogenic” gold deposits. Carles (2000) also argued that the potassium enrichment would represent a typical characteristic of lode gold deposits in amphibolite facies conditions, according to Groves (1993).

The vein-hosted mineralization would be part of a second mineralizing event, or stage, and it is interpreted as a typical “orogenic” gold type by Carles (2000) and Goodman et al., (2005). It would have been caused by hydrothermal fluids focused into the wall rocks of the felsic dikes, and within deformation zones. Gold would have been either remobilized from previous stage concentrations or introduced from a new source and would have precipitated along with quartz-sulfide veins accompanied by sericitic alteration (Goodman et al., 2005).

8.1 Discussion – Current Genetic Models

The close spatial relationship between gold and copper mineralization and the porphyritic intrusions in the Troilus deposit are also described in a series of other large Archean gold deposits. Some of these deposits, such as the Canadian Malartic and the McIntyre, share, at least in part, similarities with porphyry and/or intrusion-related gold deposits and could be genetically related to the porphyritic intrusive host rocks (De Souza et al., 2017; Mason and Melnik; 1986, Melnik-Proud 1992; Brisbin 1997 in Dubé et al., 2017). At the same time, a strong structural control of the main ore zones is observed, commonly associated with hydrothermal alteration typical of greenstone-hosted gold deposits (Groves, 1998, Poulsen, 2000; Dubé and Gosselin, 2007), which led to the interpretation that, at least in part, gold had been introduced to the system syn-main deformation phases.

Two distinct styles of mineralization are described in the Troilus deposit in terms of metal content, hydrothermal alteration, and host structures. This combination of mineralization style represents evidence of multiple stages of gold mineralization, an early magmatic-hydrothermal event followed by syn-deformational gold input and remobilization. The Troilus deposit is still poorly understood, and most of the current interpretations lack clear evidence to determine whether the age and nature of the distinct styles of mineralization (Diniz, 2019). However, it seems that at least the disseminated style of mineralization observed in Troilus, associated with a strong biotitic alteration, would have formed by magmatic-hydrothermal processes (Diniz, 2019).

9 EXPLORATION

The early exploration and development of the Project is described in Section 6. Since the acquisition of the Project, Troilus compiled historical exploration and drilling data and carried out several field mapping and prospecting programs on the Project and the Troilus Frotêt Project areas.

9.1 Exploration Review (Pre-2018)

A review of all the lithogeochemical data by Inmet indicated that a large halo with gold values greater than 200 ppb is present around Z87 and J Zones. Compilation of drillhole data for holes drilled away from the Troilus deposit has shown that there are multiple holes with gold values greater than 200 ppb over 10 m. Systematic drilling of all these zones was undertaken by previous owner companies between 1997 and 2004. Some exploration drilling was completed during this period around the old mine; however, mineralization of the continuity and grade of the main zones were not found.

In 2000, a 500 m long anomalous gold envelope, named the SW Zone, with similar characteristics to Z87 was discovered at the southwest end of the Troilus diorite. Several drillholes were drilled in early 2005 using Ingersoll Rand DML downhole hammer drill rigs (DML) to investigate the potential of having near surface mineralized material that could be mined and trucked to the Troilus mill.

9.2 Troilus (2018 to Present)

Field mapping and prospecting work in 2018 and 2019 provided support for the Troilus team to improve the understanding of the lithological and structural controls on gold mineralization across the Property and confirmed the overall potential for extending the current known limits of the main mineralized zones.

The field exploration programs on the north-eastern half of the Property (formerly Troilus North), were to evaluate the overall mineralization potential along the trend from the known deposits and to the northeast. The field exploration included geological mapping, soil geochemistry sampling and channel sampling.

In 2019 and 2020, field mapping and prospecting work focused on the Beyan gold zone, which corresponds to the SW extension of the Troilus Gold corridor, as well as at Testard in the southern part of the Troilus Frotêt property. This led to the discovery of numerous high grade gold showings and mineralized boulders confirming the potential of the Troilus Gold corridor to host gold mineralization to the southwest and of the potential to find high-grade gold mineralization along other major structures (for instance Testard).

In 2021, after completing the purchase of Urban Gold, prospecting and field mapping focused on the Bullseye-Freegold area and it continued at Beyan, extending the work done in 2020 to the south and southwest to the Cressida target. A soil geochemistry survey was completed at both targets. The other zones worked were in the central and southern part of the belt including the Bullseye-Freegold and Testard zones. After the completion of the field campaign, two drilling programs were completed during the winter at the Cressida Target and Testard Target.

In 2022, results from the previous programs led to work being initiated at Cressida and Freegold-Bullseye Targets including completing a higher resolution soil geochemistry survey, field mapping, prospecting and a second drilling program at Cressida. The field mapping and prospecting also focused on the Freegold and the newly outlined Pallador areas with some targeted traverses in the south and east of the Troilus Frotêt property. At the Pallador Target, a five-drill hole program was completed. These drill holes tested a soil anomaly that coincides with high-grade boulder samples. At the Testard Target, a single drill hole was completed, also targeting a high-grade gold anomaly. A large scale till sampling survey was completed that focused-on counting of gold grains in the southern part of the Troilus Frotêt property in 2022.

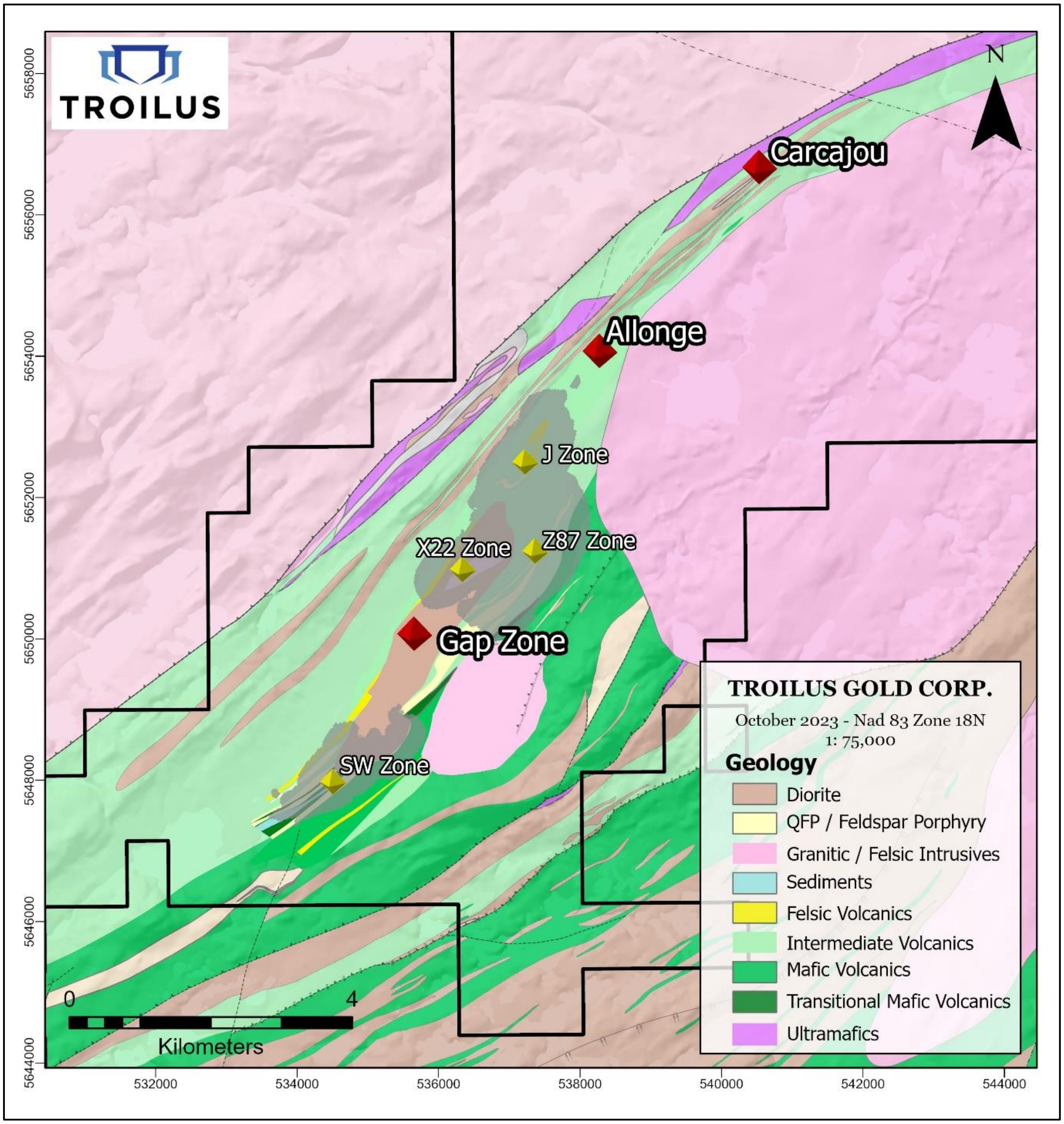
In early 2023, ground IP surveys were completed on two grids and an airborne VTEM survey was flown in the Pallador area. A mapping and sampling program was completed in the Pallador area. An additional two drill holes were completed to test the IP anomalies. Mapping and sampling program also focused to the northeast of the Troilus Gold property around the Parker granite.

9.2.1 Troilus Gold Property

Allongé and Carcajou Targets

The Allongé target is situated adjacent to the northeast of the J Zone, with the Carcajou target a further 5 km northeast along strike (Figure 9-1). Prospecting done by Inmet in the 1980s on the Holmstead showing, located between Allongé and Carcajou, reported two grab samples over 5 g/t Au, situated 1 km east of the north-northeast trending Lac Allongé. The Carcajou showing reported a grab sample of 8 g/t Au.

Figure 9-1: Location Map of Exploration Targets – Troilus Gold Project



Source: Troilus (2023)

Troilus conducted a prospecting and mapping program in the Allongé Zone in late 2018, collecting 172 samples for assay. Highlights included a 110 g/t Au grab sample found 1 km along strike from the J Zone pit and 4.33 g/t Au from channel sampling 1.8 km northeast of the J Zone, among other high-grade samples. The success of this surface exploration led to the planning of an 11-hole; 1,995 m diamond drill program undertaken in March 2019. The holes were planned on three sections each spaced 500 m apart, extending 1.5 km northeast of the J Zone. Wide lenses of low-grade gold mineralization were intercepted, extending the known mineralized corridor to the northeast by 1 km. The most promising results were found in drillholes TLG-ZJ4N19-122 and TLG-ZJ4N19-123, with 0.47 g/t Au over 22 m and 0.33 g/t Au over 66 m, respectively. The holes further to the north were terminated by the Parker pluton, and only reported sporadic low-grade gold assays.

A 12-hole, 2,857 m diamond drill program was conducted in the summer of 2021 by Troilus across four claims with the aim of extending mineralization in the J Zone further to the northeast. Four holes were drilled in Allongé totalling 1,452 m and eight holes were drilled in Carcajou totalling 1,405 m. All four holes in Allongé intercepted several lenses of gold mineralization between 0.3 - 0.5 g/t over 1 – 16 m. Notably, hole ALG-21-003 returned one result of 8.6 g/t over 1 m in a pyritic shear zone. The drilling at Carcajou intercepted the same intermediate volcanic package present in the J Zone but hit significantly more barren granite of the Parker pluton than expected. As a result, only one hole, CAR-21-006, returned any gold assays above 0.3 g/t.

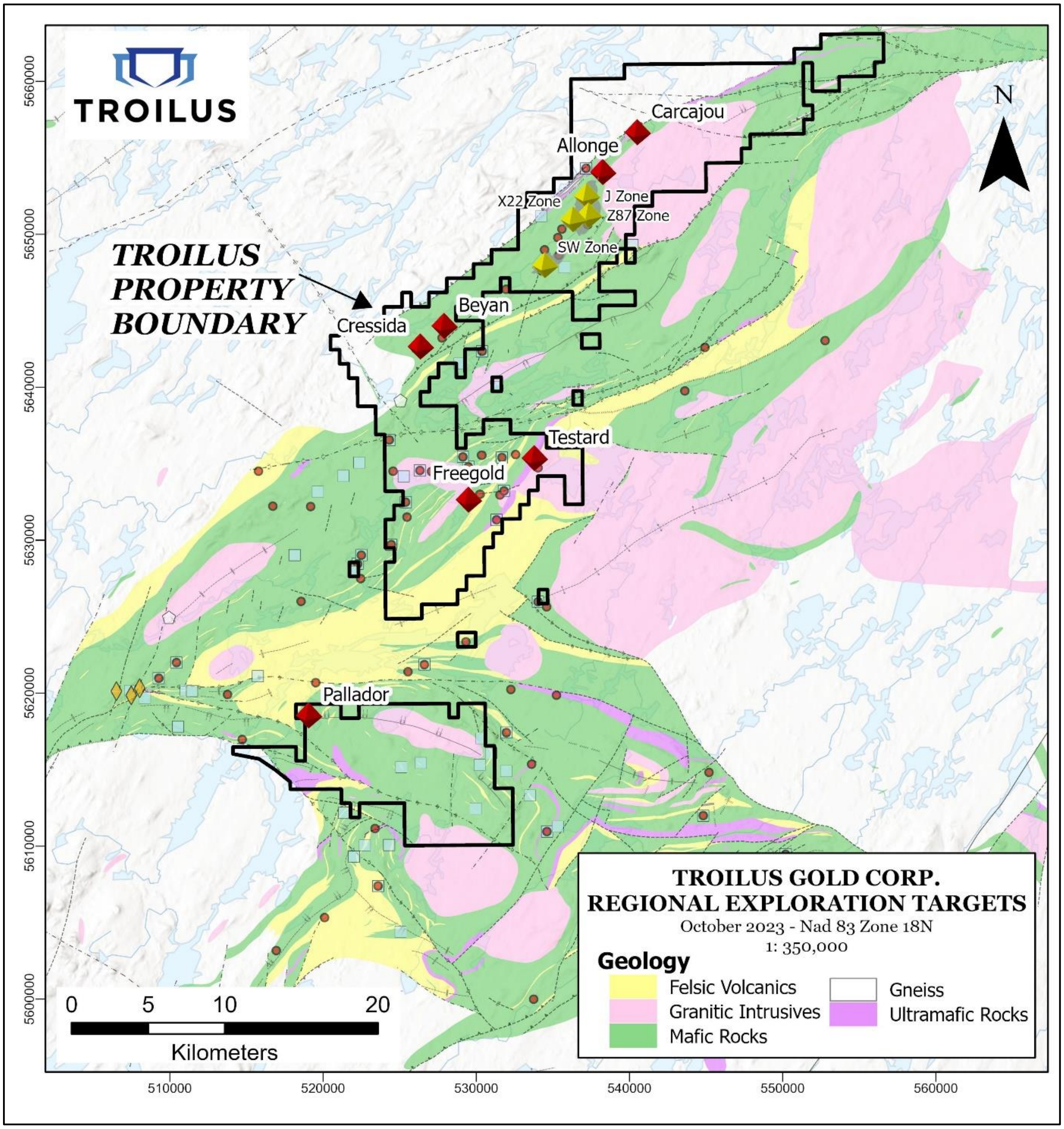
9.2.2 Troilus Frotêt Property

Following a major compilation of historical data and based on field observation, Troilus re-evaluated the potential of the entire Frotêt-Troilus segment of the Frotêt-Evans greenstone belt by acquiring a major land position called Troilus-Frotêt Property (Figure 9-2).

Several types of mineralization are present on the Property including:

- Volcanogenic Massive Sulphide (VMS)
- Orogenic gold and shear-hosted polymetallic (Au-Cu-Ag-Zn-Mo) quartz veins
- Lithium and rare metals (Ta, Sn, Rb, Cs, Be, Mo) bearing pegmatites
- Multi-element (Mo-Bi-W-Ag-Cu-Zn) quartz viens

Figure 9-2: Location Map of Exploration Targets – Troilus Frotêt Project



Source: Troilus (2023)

In June 2020, Troilus completed a preliminary field exploration program applying a new regional structural and geological model to the recently expanded Troilus-Frotêt property. This property is situated to the south of the main mineralized zones of the Troilus deposit.

During the summers of 2020 and 2021, Troilus completed two airborne high-resolution magnetic geophysical surveys that covered a total of 23,000 line-km and 4,768-line km respectively over the entire Troilus Frotêt area.

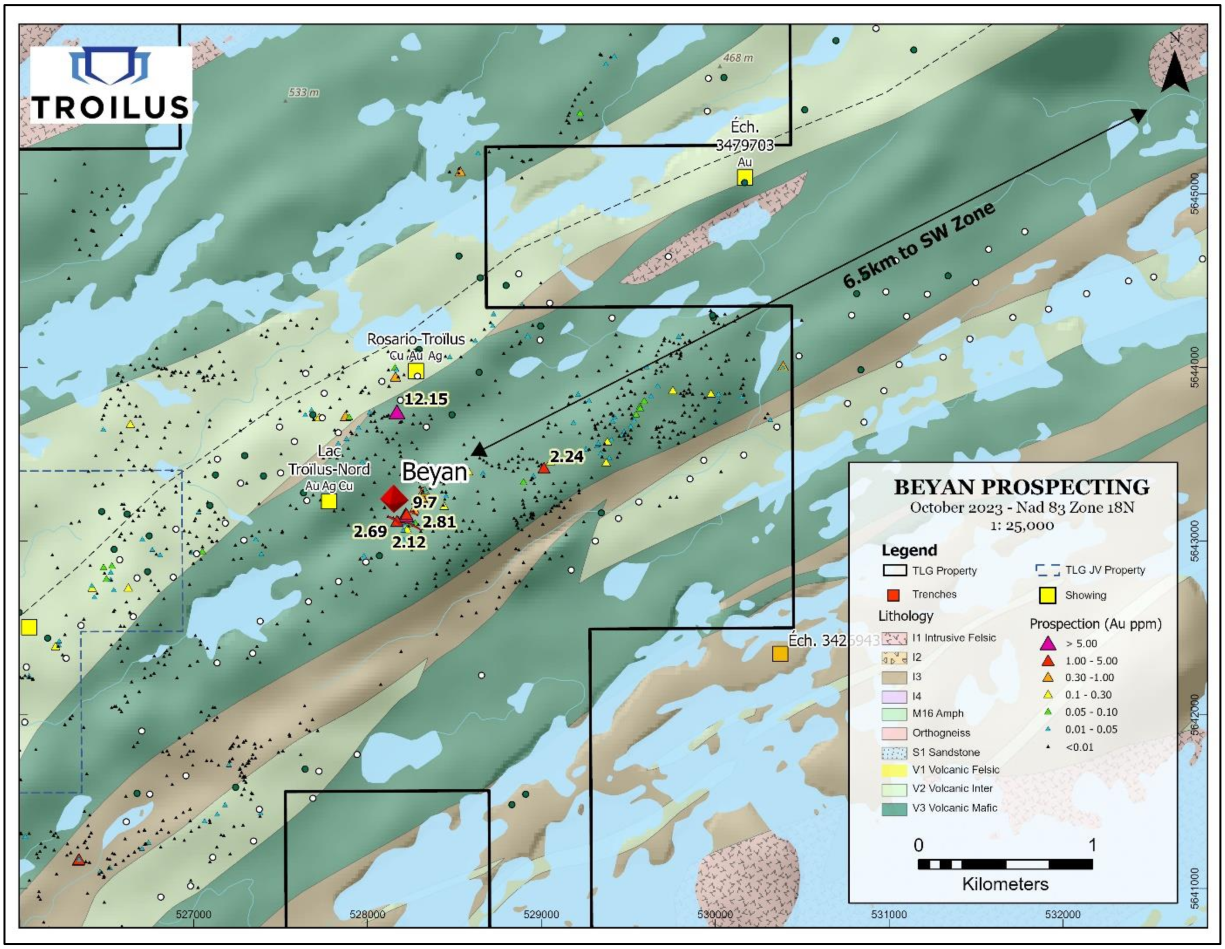
The airborne surveys were carried out by Prospectair Geosurveys Inc., based in Gatineau, Québec. Troilus also completed several B-horizons soil geochemistry surveys on the Troilus-Frotêt property for a total of 9,780 samples. The soil surveys were carried out by SL Exploration. During the field seasons 2021 and 2022, the Troilus exploration team proceeded with surface mapping and prospecting of targeted areas of its 1,420 km^2 property, leading to the discovery of numerous gold-silver and base metals mineralization zones.

Beyan Target

Initial bedrock mapping and boulder tracing along the Route de la Mine North Block claims, situated approximately 8 km southwest and along strike of the SW Zone led to the discovery of the Beyan Gold Zone near the Rosario-Troilus (best values of 3.5% Cu, 1% Zn, 12.5 g/t Au, and 161.7 g/t Ag in channel samples) and Lac Troilus-Nord showings (11.4 g/t Au and 0.94% Cu over 0.5 m) (Figure 9-3). To date, 25 outcrop grab samples have returned anomalous gold values greater than 0.1 g/t Au with the best results returning 9.7 g/t Au and 32.5 g/t Ag. A total of 14 grab samples from the Beyan Gold Zone have been collected from outcrop and can be traced on strike over 225 m. This gold zone is part of a larger gold-bearing boulder field, identified by Troilus, characterized by several boulders containing gold and silver values up to 2 g/t Au and 4.9 g/t Ag. These boulders were found over a distance of 2.5 km.

On the Beyan Zone, the Troilus geological team opened four trenches, totalling 400 m, perpendicular to the regional stratigraphy to gather more information on the geology and structure of the zone. Field observation and mapping showed that the Beyan Zone is characterized from SE to NW: an amphibolitized bimodal volcanic sequence, a mafic intrusive complex, an intermediate volcaniclastic sequence, similar to that observed in the J Zone, and finally a mafic volcanic sequence.

Figure 9-3: Troilus Frotêt Project – Beyan Target



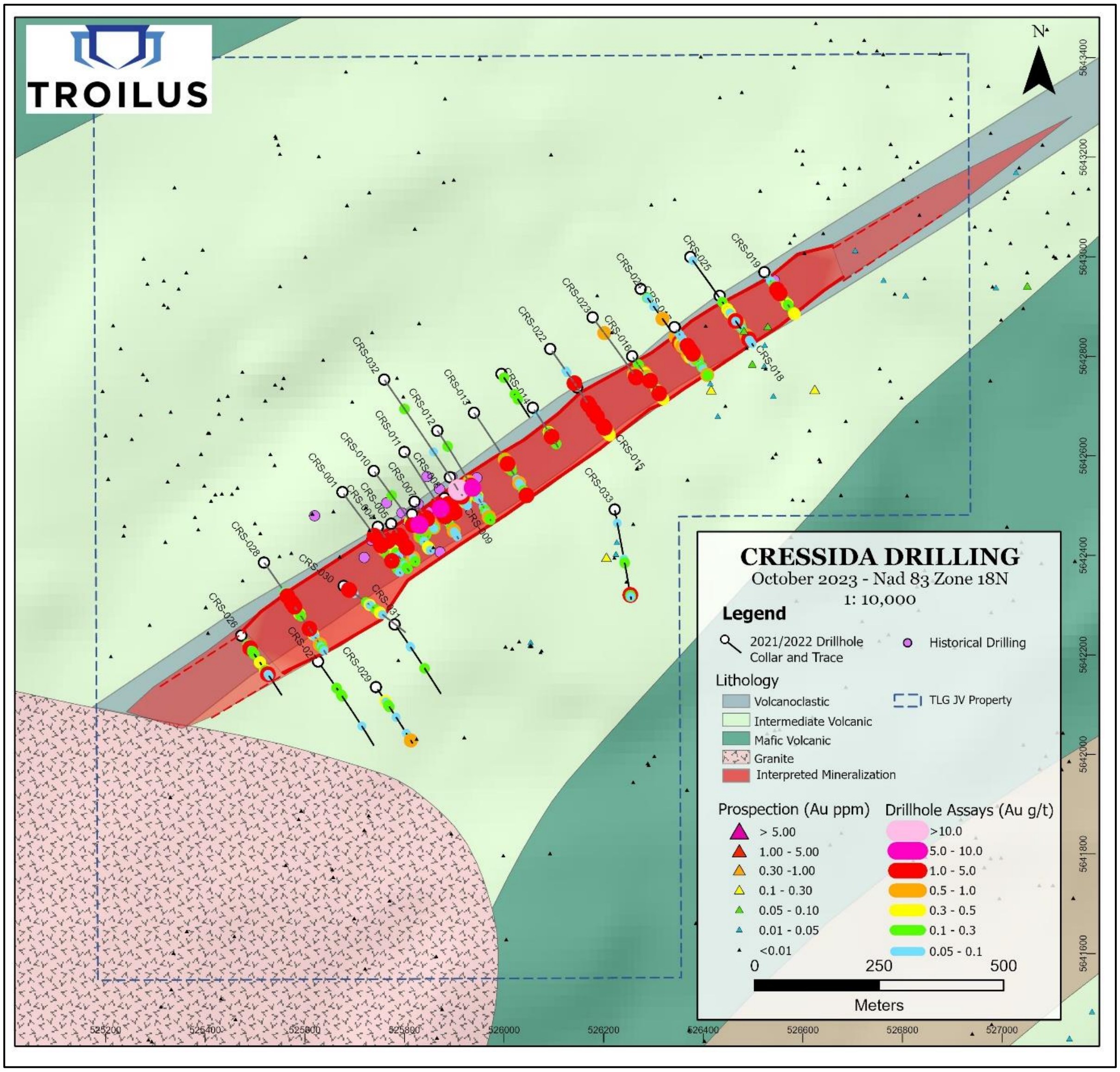
Source: Troilus (2023)

Cressida Target

The Cressida claim block is situated approximately 14 km southwest of the Troilus deposit and 2.5 km southwest of the Beyan Gold Zone (Figure 9-4). The claim block consists of five mineral claims held under a 50:50 joint venture agreement with Argonaut Gold Ltd. (Argonaut Gold), with Troilus being the operator.

Geological mapping, trenching, and geophysical surveys were conducted by Muscocho Exploration Ltd. in the late 1980s and followed up by two diamond drill programs totalling 2,416 m over 31 drillholes. The programs targeted two parallel highly conductive magnetic anomalies identified by VLF-EM surveys. They returned high grade gold values including 0.22 ounces per ton (oz/t) (7.5 g/t) over 1.8 m and 1.65 oz/t (56.6 g/t) over 0.47 m, inside a wider envelope of lower grade mineralization up to 0.99 g/t Au over 44.57 m. However, no follow up work was done to extend the 300 m strike length of the target.

Figure 9-4: Troilus Frotêt Project – Cressida Target

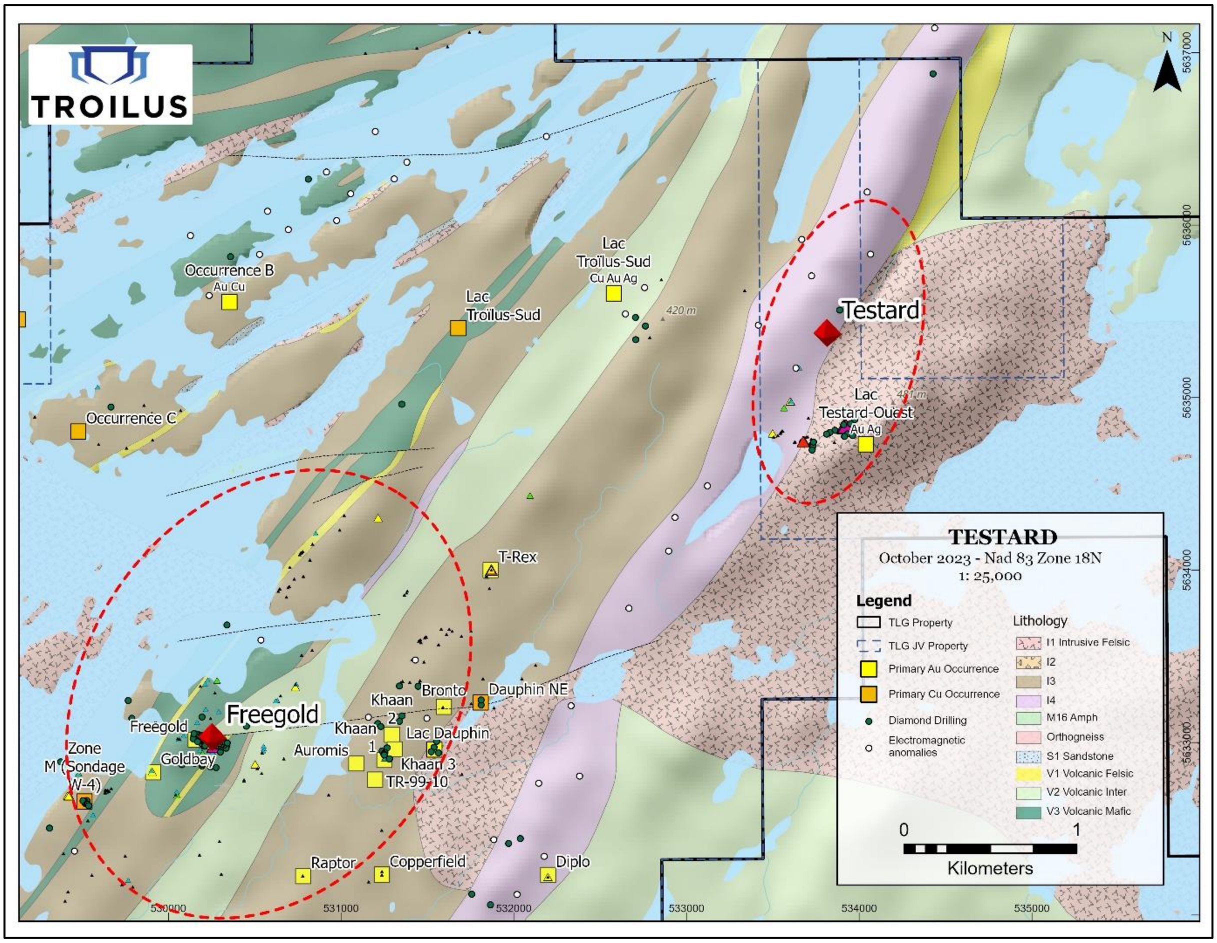


Source: Troilus (2023)

Testard Target

The Lac Testard-Ouest (Testard) showing (Figure 9-5), was discovered in 1989 by prospecting by Flanagan McAdam & Company (GM47325). The discovery was followed by a 16-hole, 1,328 m drill program conducted by Muscocho Explorations Limited, which confirmed the presence of mineralized quartz veins at depth and along strike from the surface showing (GM47326).

Figure 9-5: Troilus Frotêt Project – Testard Target and Freegold Target



Source: Troilus (2023)

In early 2020, Troilus acquired the Testard claim area and completed a surface mapping, prospecting, and outcrop stripping program over the course of that summer.

Outcrop sampling returned significantly high gold grades within the Frotêt-Evans Greenstone Belt with values returning up to 203 g/t Au, 2,440 g/t Ag and 4.37% Cu. Structural field ma. ng and interpretation of the airborne magnetic and IP data led to a new geological and structural model for the area on which the Testard drill program was built. The drill program a. d to test different structures in the area that had the potential to carry gold mineralization, while also testing extensions of the high-grade mineralization below surface at the main showing.

Freegold-Bullseye Target

The claims from the Freegold-Bullseye zone were acquired during the acquisition of Urban Gold in 2021, the claims of this zone are held under a 50:50 joint venture agreement with Argonaut Gold, with Troilus being the operator (see Figure 9-5). The geology consists of NE-SW-trending volcano-sedimentary sequences, intercalated with mafic and ultramafic sills, and intruded by the tonalitic

Testard and Lac Troïlus-Sud stocks. The geologic setting is primarily prospective for gold, silver, and base metals over several different deposit styles including orogenic gold (Au, Ag, Cu) and volcanogenic massive sulphide (Cu, Zn, Au, Ag).

Historically the area was worked for gold and base metals mineralization, more specifically the eastern shore of the Troilus Lake. The Lac Dauphin showing (Au-Ag-Cu-Zn-Mo) was discovered by Dauphin Iron Mines Ltd. in 1958. The Troilus Freegold showing was discovered in 1966 - 1967 by Troilus Mines Ltd. This zone, which contains visible gold, represents one of the most worked showings of this part of the belt. It was drilled during multiple campaigns, and resources were estimated by SOQUEM in 1999 at 15,880 t grading 16.45 g/t Au (SOQUEM Inc. used the GM 56807 and 57907 for the estimate, which is not 43-101 compliant, source SIGEOM website). Located nearby, the M-Zone represents a vein hosted copper-rich type of mineralization. In 1999, SOQUEM completed a major stripping campaign in the area, especially east and north of Troilus Freegold zone. As suggested by Bellavance (1999) following the exploration work performed by SOQUEM Inc. in 1998 and 1999, gold would preferentially occur in an east-west-trending and ~1.5 km wide deformation corridor. This corridor has been historically highlighted by multiple exploration and drilling programs that were accomplished since the 1950s, and the discovery of several gold and copper occurrences such as Freegold Zone, M Zone, the Lac Dauphin showing, among others.

In 2019, prospecting work led by Laurentia Exploration on behalf of Urban Gold led to the discovery of new Au-Cu-Ag (Zn) showings confirming the potential of the Freegold-Bullseye Zone.

In 2021, a prospecting, stripping, and channelling campaign was carried out by Troilus with Dahrouge Geological Consulting Ltd. from mid-September to mid November 2021, which included a structural interpretation of the area. This led to the development of a new model for gold mineralization and the discovery of new prospective zones.

The Freegold area is characterized by a complex network of D1 and D2 deformation zones and by numerous precious and base metal showings highlighting the high economic potential of the zone.

The M Zone showing (E-529400 m, N-5632655 m) was drilled in 1996 and 1997 by Muscocho Exploration Ltd. with interesting intercepts such as 3.00% Cu and 37.5 g/t Ag over 5.1 m (DDH W-2); 2.00% Cu and 31.21 g/t Ag over 3.0 m (DDH W-4); and 2.52% Cu and 26.5 g/t Ag over 3.7 m (DDH W-6) (GM20679; GM57907). In 2019, Urban Gold returned and drilled along IP anomalies without finding these intercepts. They noted that the location of the historical drillholes was not found (1730-2019_Urban Gold_Report-Drilling). The drillholes completed by Urban Gold near the historical M Zone (DDH UTB-19-10, -13, and -14) have shown a wide variety of rock types such as gabbro, basalt, and felsic to intermediate volcaniclastics. Old blast holes near the drilled zone were found during this program. These holes exposed a silicified shear zone at the contact between a gabbro and a basalt oriented at N300 with a steep dip. The shear and mineralized vein appear to be emplaced preferentially along the contact between the two lithologies, the vein is mineralized with chalcopyrite and pyrite with malachite and azurite alterations.

The Lac Dauphin showing (or TRM-99-02; E-531531 m, N-5632929 m) was discovered in 1958 by Dauphin Iron Mines LTD and corresponds to a NE-SW shear zone at the contact between tonalite of the Testard stock and a strongly sheared and altered mafic rock. Best values were returned by channel sampling and consists of up to 3.2 g/t Au, 64.8 g/t Ag, and 0.51% Cu over 3 m (GM 57907), 6 g/t Au, 257 g/t Ag, 6.64 % Cu and 1.895 % Zn over 50 cm (sample C553928) and 6.55 g/t Au, 100 g/t Ag over 50 cm including 0.48 % Cu and 0.7 % Zn (sample C553881). The shear-zone follows the contact with the tonalite which changes direction and at this inflexion point a secondary shear develops within the tonalite, and a decametric mineralized quartz vein with tension veins follows this second-order shear. Whereas the less competent mafic rock has been mylonitized and completely metasomatized and has been transformed into a chlorite-carbonate-fuschite schist at the contact with the tonalite. Mineralization in the veins consists of disseminated to massive chalcopyrite, pyrite, and sporadic molybdenite, sphalerite, bornite, malachite and native silver.

Pallador Target

The Pallador zone is located just south from the Regnault deposit (Kenorland-Sumitomo JV ground) and 35 km south from the Troilus mine. The area was worked in 2002 by SOQUEM which carried out an exploration program for Pt-Pd mineralization in mafic-ultramafic sills (GM 59962) which included mapping, trenching, sampling together with IP and magnetic surveys. This led to the recognition of a PGE-enriched zone of 40 m wide by 550 m long with values above 100 ppb Pd-Pt

In 2018, Urban Gold conducted a 4-hole drilling program to test some EM inputs and they intercepted intervals with low-grade Cu and Zn mineralization typical of a VMS system (GM71292).

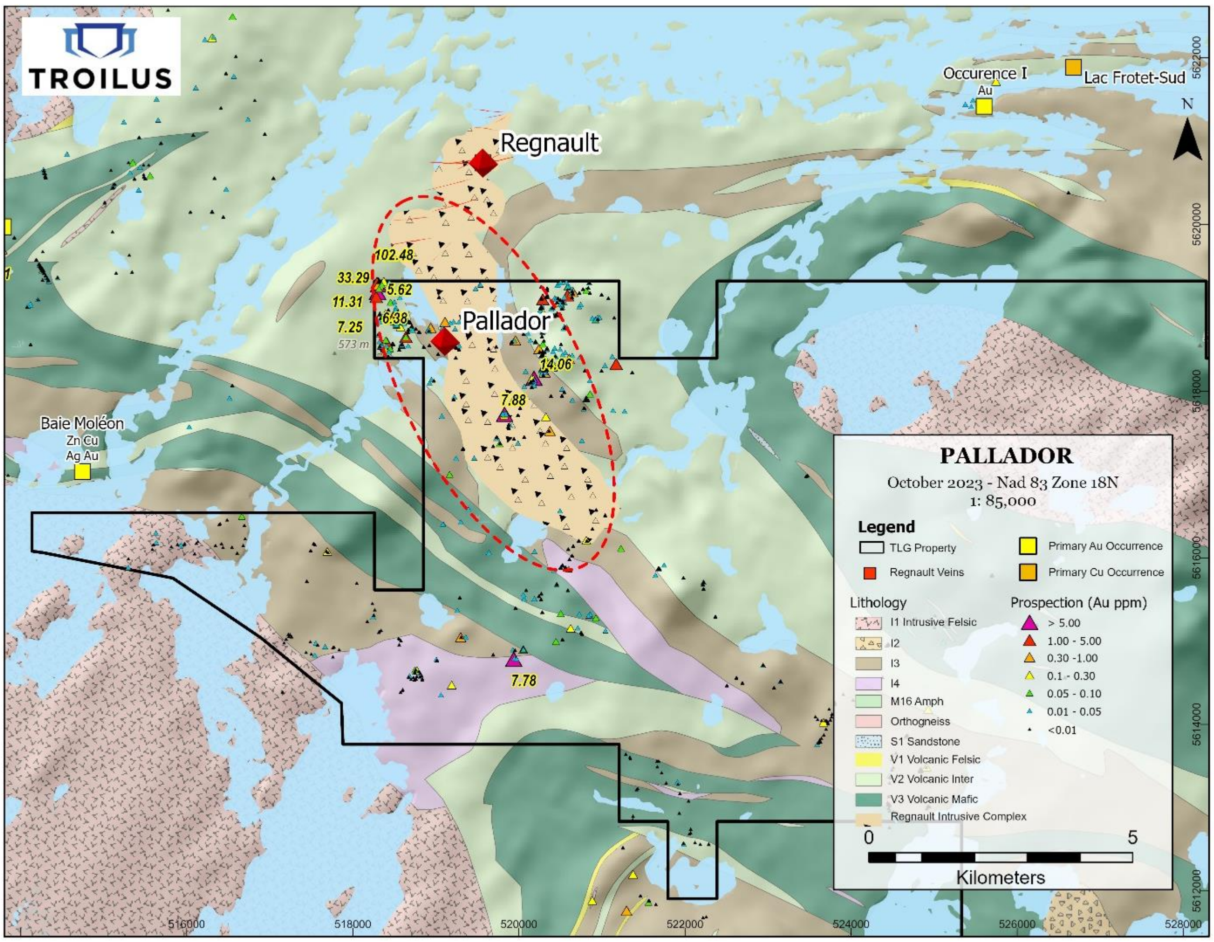
In 2020, after the discovery of the Regnault gold prospect by Kenorland (discovery hole 20RDD007 with 29.08 m @ 8.47 g/t Au and 12.23 g/t Ag, including 11.12 m @ 18.43 g/t Au and 25.93 g/t Ag), Prospectair Geosurveys Inc. conducted a high-resolution airborne magnetic survey of the area for Urban Gold. This survey was followed by a prospecting and soil sampling campaign led by Laurentia Exploration leading to the discovery of numerous gold-bearing boulders and outcrops (Figure 9-6). Using rock and soil assays, Urban Gold decided to do an IP survey in the northwest part of the property and during the winter 2020 – 2021 drilled 10 holes for a total of 2,454.5 m. The drillholes located in the east intercepted gold mineralization near surface and at depth with the highest result being 4.75 g/t Au over 2.05 m including 19.24 g/t Au over 0.5 m with native gold grains in drill hole UPR-21-09 from 240.95 m.

Between 2020 and 2022, soils surveys were conducted totalling 2,637 samples as the grids were established around anomalies in subsequent years. The most prominent anomalies of gold in soil originate from an area referred to as Rocket. Soil anomalies coincide with boulders of gabbro containing up to 5% pyrite and returning results up to 32.2 g/t Au and 25.4 g/t Au. As a result, a 5-hole drill program was conducted in 2022 to test the geology below the pervasive cover and target interpreted magnetic features proximal to the up-ice origin of the mineralized boulder fields. The highest results returned 2.45 g/t Au over 1 m and 4.43 g/t Au over 1 m from the same drill hole (RCK-22-004). Mineralization was associated with sheared and silicified gabbro containing intermittent quartz veining and up to 5% pyrite locally.

In 2022, a potentially new volcanic massive sulphide (VMS) trend was also discovered in the Pallador sector. The Copper Bay showing (0.44% Cu, 0.33% Zn, 2.6 g/t Ag) and Branphil showing(3.94% Cu, 8.4 g/t Ag, 35 ppb Au) are situated roughly 750 m apart on a 1,200 m conductor that was followed up using beep-mat and VLF. The conductor was locally excavated by hand for sample collection and the Branphil showing was stripped by portable excavator and washed to expose approximately 600 m^2 of bedrock. Channel samples were collected across the mineralized horizon.

In 2023, an airborne VTEM survey totalling 248-line kilometres was flown over the Copper Bay-Branphil trend and is inclusive of other showings in the area (Rhyolite and Monique). These surveys were used to generate drill targets in their respective areas. In September 2023, a two drill holes were completed, totalling 635 m, to test chargeability anomalies at Rocket. Drilling intersected the same highly magnetic gabbro encountered in the 2022 drilling at approximately 1 km along strike to the northwest. The best result from this drilling was 2.93 g/t Au over 1 m from a locally sheared gabbro with disseminated and vein-hosted pyrite. In the winter of 2023, ground IP surveys were completed on two grids in the Pallador sector totalling 30.025- and 16.15-line kilometres over Rocket and Copper Bay-Branphil, respectively.

Figure 9-6: Troilus Frotêt Project – Pallador Target

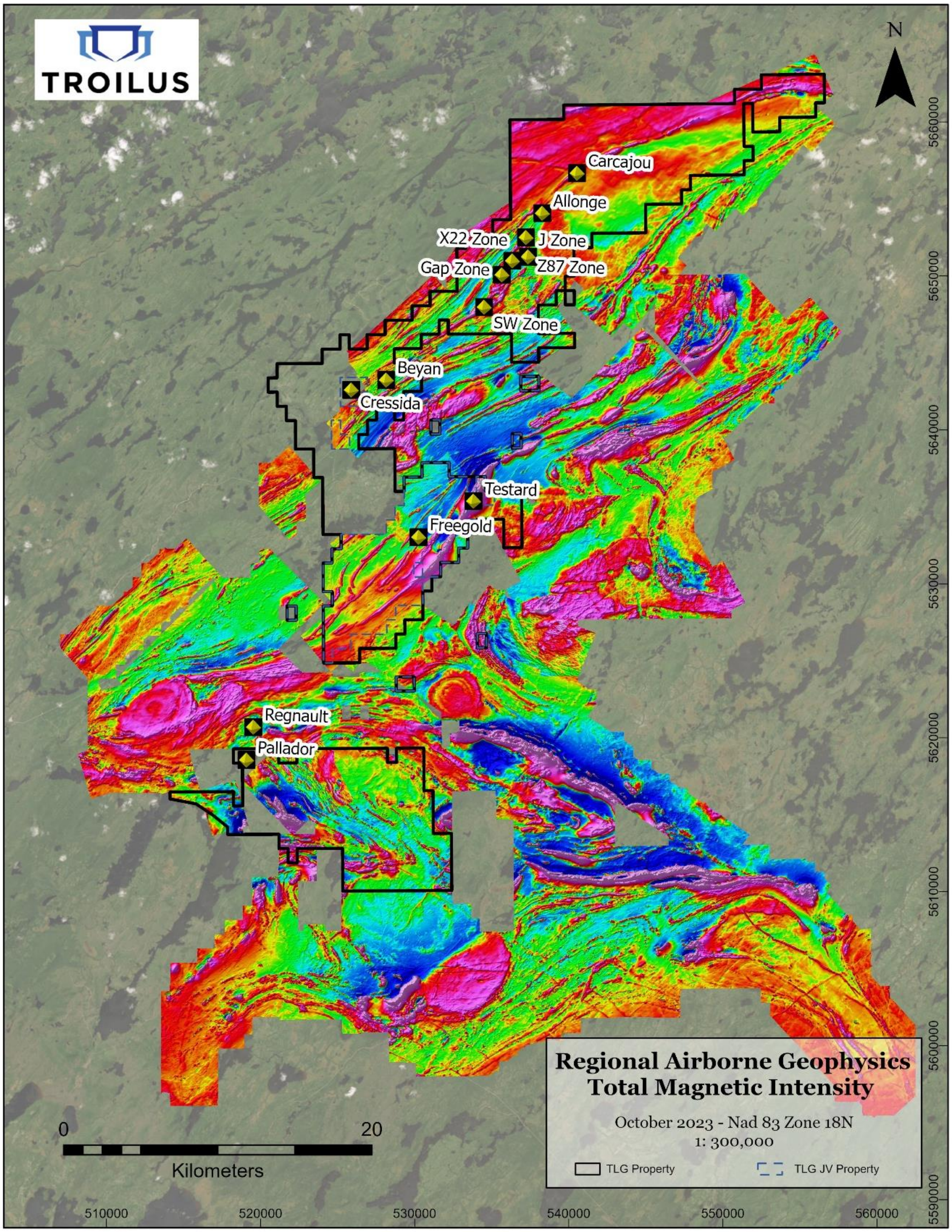


Source: Troilus (2023)

9.3 Geophysical Surveys

During the summers of 2020 and 2021, Troilus completed two airborne high-resolution magnetic geophysical surveys that covered a total of 23,000 line-km and 4,768-line km respectively over the entire Troilus Frotêt area (Figure 9-7).

Figure 9-7: Regional Airborne Geophysical Survey – Total Magnetic Intensity



Source: Troilus (2023)

In the Fall and Winter of 2020-2021, several ground geophysical surveys were conducted along the Troilus trend. These surveys were IP and Polar L Dipole P and were completed at Carcajou, Southwest (which included parts of the Gap Zone) and Beyan-Cressida. These same geophysical surveys were also completed at Testard.

Table 9-1 provides the technical specifications of the ground geophysical surveys in 2020 to 2021.

Table 9-1: Summary of Technical Specifications for Geophysical Surveys, 2020 – 2021

Target	Fieldwork	Survey Length	Period
Carcajou	Line cutting IP, PLDP (a=25m, n=1 to 20)	105.075-line km 34.025-line km	Nov 1-21, 2020
Southwest	Line cutting IP, PLDP (a=25m, n=1 to 20) IP, PLDP (a=55m, n=1 to 10)	105.075-line km 34.025-line km 35.05-line km	Oct 16-30, 2020
Beyan	Line cutting IP, PLDP (a=25m, n=1 to 20) IP, PLDP (a=50m, n=1 to 10)	105.075-line km 34.025-line km 35.05-line km	Phase I: 22 Sep – 15 Oct, 2020 Phase II: Feb 16-28, 2021
Testard	Line cutting IP, PLDP (a=25m, n=1 to 10)	22.45-line km 20.045-line km	Oct 16-30, 2020

In 2023, an airborne VTEM survey totalling 248-line kilometres was flown over the Copper Bay-Branphil trend and is inclusive of other showings in the area, for example, Rhyolite and Monique. Ground IP surveys were also completed on two grids in the Pallador sector totalling 30.025- and 16.15-line kilometres over Rocket and Copper Bay-Branphil showings, respectively.

9.4 Petrology, Mineralogy, and Research Studies

In 2019, Troilus engaged Dr. Neil Banerjee of Western University in a research agreement. Since the onset of the agreement, Dr. Banerjee has had multiple geology students work on the Troilus deposit. Under Dr. Banerjee, Tavis Enno has completed an undergraduate thesis on the Troilus deposit titled, "Alteration and Mineralization of the Southwest Zone at the Troilus Gold-Copper Project, Québec: Implications for a Revised Genetic Model" and is currently working on a master's thesis. Mac Valliant completed a research project titled, "Petrographic and Geochemical Analysis of the Troilus Gold-Copper Deposit, Québec" and Adrienne Iannicca completed a research project titled, "Fluid Inclusion Study and Gas Analysis of Quartz-Carbonate Veins from the Troilus Gold-Copper Deposit."

In 2020, Troilus engaged Ultra Petrography and Geoscience Inc. to complete a petrographic report on 13 samples taken across the Troilus deposit.

9.5 Exploration Potential

Within the Freegold-Bullseye block there are several gold-in-soil anomalies that exist ~5.5 km southwest of Testard along the same prospective corridor. Anomalies coincide with D1 and D2 structural trends, as interpreted from airborne magnetics. This area is referred to as Katana and the source of these anomalies has yet to be thoroughly investigated.

10 DRILLING

10.1 Drilling Summary

Since 1986, there have been several drilling programs completed on the Property. There was no drilling on the Property from 2008 to 2017, and Troilus' drill programs were completed from 2018 to 2023. Table 10-1 summarizes the diamond drilling programs completed on the Property to date.

Troilus completed 91 drillholes totalling 37,510 m in 2018; 87 drillholes totalling 38,172 m from 2019; 43 drillholes totalling 21,185 m in 2020; 193 drillholes totalling 84,112 m in 2021; and 161 drillholes totalling 78,775 m in 2022; and 115 drill holes totalling 38,780 m in 2023. Most of the 2018 and 2019 drillholes targeted the Z87 Zone and the J Zone at depth and along strike. 2020 and 2021 were mainly focused on developing the Southwest Zone (SW Zone). Drilling in 2022 was focused on SW Zone and additional drilling at Z87. Drilling in 2023 was primarily focused on the southwest extension of the Z87 Zone, that is, development of the X22 Zone; and in the Connector area between the J Zone and the Z87 Zone.

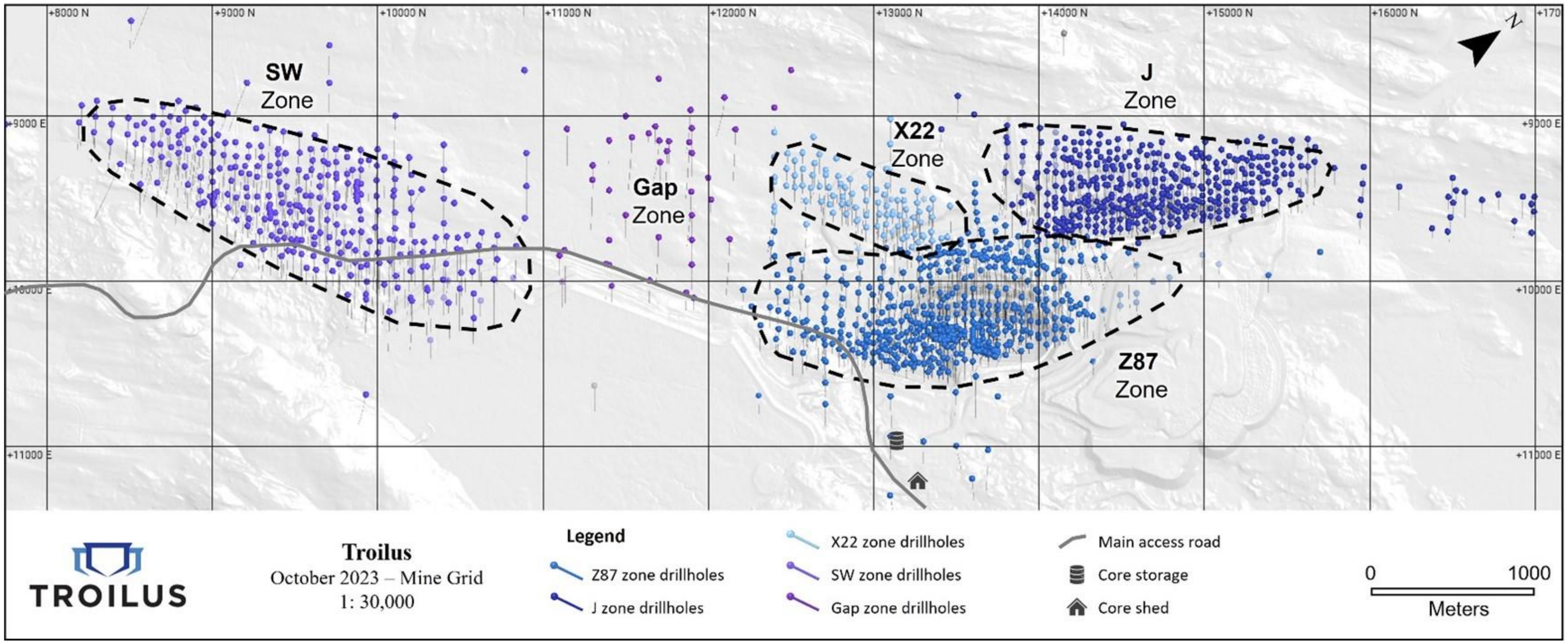
Table 10-1: Summary of Drilling

<table>
<tr><th>Year</th><th>Contractor</th><th>Core Size</th><th>No. Holes</th><th>No. Metres</th></tr>
<tr><td>1986-1989</td><td>Morissette Diamond Drilling</td><td>BQ (36.5 mm)</td><td rowspan="14">698</td><td rowspan="14">134,068</td></tr>
<tr><td rowspan="3">1990</td><td>Morissette Diamond Drilling</td><td rowspan="3">NQ (47.6 mm)</td></tr>
<tr><td>Benoit Diamond Drilling</td></tr>
<tr><td>Chibougamau Diamond Drilling</td></tr>
<tr><td rowspan="2">1991-1993</td><td>Benoit Diamond Drilling</td><td rowspan="2">NQ</td></tr>
<tr><td>Chibougamau Diamond Drilling</td></tr>
<tr><td rowspan="2">1995</td><td>Benoit Diamond Drilling</td><td>NQ ("KN" holes)</td></tr>
<tr><td>Morissette Diamond Drilling</td><td>BQ ("TN" holes)</td></tr>
<tr><td>1997</td><td>Chibougamau Diamond Drilling</td><td>NQ ("KN" holes); BQ ("TN" holes)</td></tr>
<tr><td>1999</td><td>Forages Mercier</td><td>NQ</td></tr>
<tr><td>2000</td><td>Chibougamau Diamond Drilling</td><td>NQ (Z87 and J4 Zones); BQ (elsewhere)</td></tr>
<tr><td>2002</td><td>Chibougamau Diamond Drilling</td><td>NQ</td></tr>
<tr><td>2003-2005</td><td>Forages Mercier</td><td>NQ</td></tr>
<tr><td>2007</td><td>Forages Mercier</td><td>NQ</td></tr>
<tr><td>2018</td><td>Chibougamau Diamond Drilling</td><td>NQ</td><td>90</td><td>37,342</td></tr>
<tr><td>2019</td><td>Chibougamau Diamond Drilling</td><td>NQ</td><td>87</td><td>37,899</td></tr>
<tr><td>2020</td><td>Chibougamau Diamond Drilling</td><td>NQ</td><td>17</td><td>6,038</td></tr>
<tr><td>2021</td><td>Chibougamau Diamond Drilling</td><td>NQ; HQ (TLG-ZSW21-212-GT); BTW (GZ)</td><td>193</td><td>84,112</td></tr>
<tr><td>2022</td><td>Chibougamau Diamond Drilling</td><td>NQ; BTW (GZ)</td><td>161</td><td>78,775</td></tr>
<tr><td>2023</td><td>Chibougamau Diamond Drilling</td><td>NQ</td><td>115</td><td>38,780</td></tr>
</table>

Notes: Inside Diameter – 36.4 mm; NQ – 47.6 mm; HQ – 63.5 mm; BTW – 42 mm

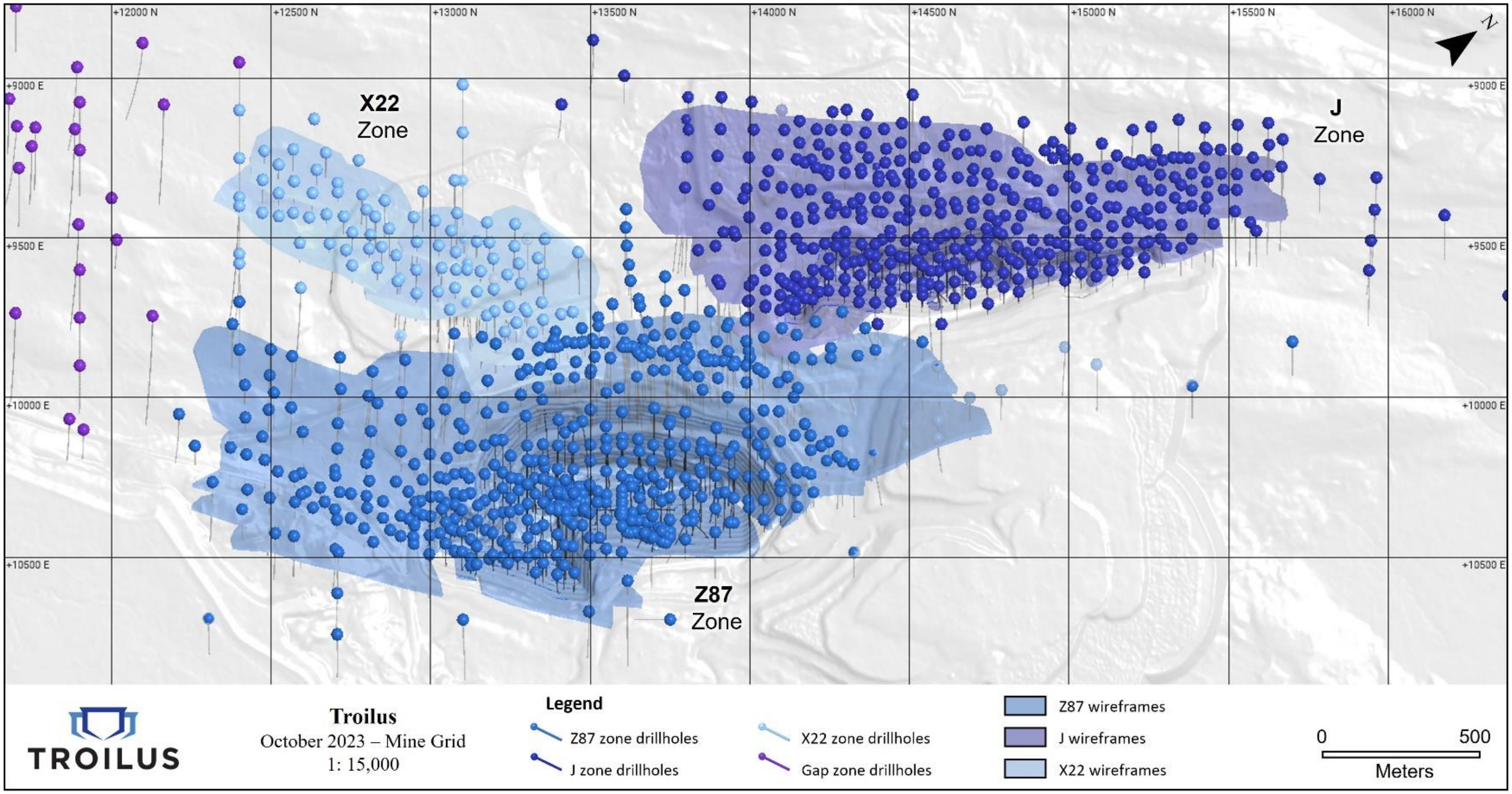
Figure 10-1 illustrates the drilling completed on the four mineralized zones of the Project, which are included in the resource drill hole data base. Figure 10-2 presents the drill hole locations on the Z87 Zone, J Zone, and X22 Zone. Figure 10-3 presents the drill hole locations on the SW Zone.

Figure 10-1: Drill Hole Map – Troilus Gold Project



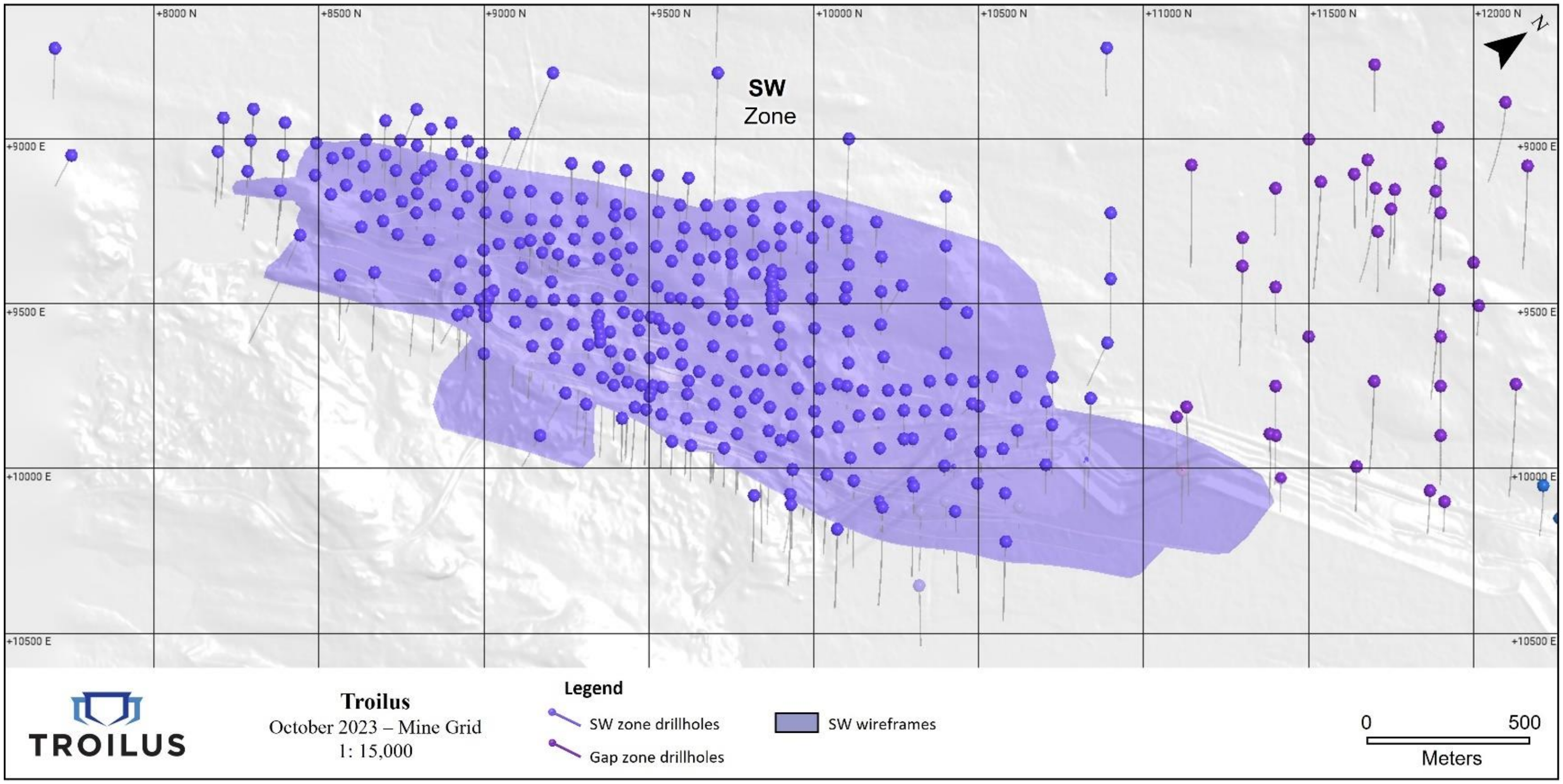
Source: Troilus (2023)

Figure 10-2: Drill Hole Location Map – Z87 Zone, J Zone and X22 Zone



Source: Troilus (2023)

Figure 10-3: Drill Hole Location Map – SW Zone



Source: Troilus (2023)

10.2 Drill Methods and Logging (2018 – 2023)

Troilus completed its own drilling on the Property between 2018 and 2020. Troilus contracted Chibougamau Diamond Drilling Ltd. (Forages Chibougamau Ltée), based in Chibougamau, Québec. All drill core was NQ size diamond drill core.

Drill rigs were set up with siting stakes and marked with the azimuth and dip. Collar coordinates were initially measured using hand-held GPS units measuring in NAD83 Datum and converted to mine grid. Once a set of drill holes, or program, is completed, drill hole collars were surveyed using a differential GPS by M. Paul Roy, a professional land surveyor based in Chibougamau. Coordinates for the drill collars are delivered in UTM NAD83 and Mine Grid. In 2021, Troilus was using the Arrow 100 series high-accuracy GPS from EOS Positioning Systems, internally. Every hole is surveyed with this GPS upon completion of the hole by Troilus personnel.

10.2.1 Downhole Surveys

From 2018 to December 2021, drillholes were surveyed downhole using either a Reflex or EZ Gyro device. A multi-shot survey was carried out from the end of each hole (Reflex by 3 m increments, EZ GYRO by 30 m increments).

From January 2022, Troilus switched instruments to the Devico DeviGyro Overshot Xpress. A single-shot test is taken daily for each hole, and a continuous test is taken at the end of every hole, at 3 m intervals. If the single-shot test determines that the hole is deviating substantially, more tests would be taken daily to monitor the progression of the hole.

10.2.2 Drill Core Logging

Troilus maintains Standard Operating Procedures for all aspects of core handling, logging, sampling, and storage. AGP has reviewed these procedures and found they meet or exceed industry practice.

Drill holes completed by Troilus are labelled as:

$$TLG{-}< zone >< year > -< number >;\ for\ example\ TLG - Z8718 - 001$$

All drill core collected was placed in 1.5 m long, three-row wooden core boxes. Meterage is marked by drillers using wood blocks with the metre depth marked in black marker every three metres. Drill core boxes are marked on the left edge and top with the drill hole number and core box number. The drill core is transported to the core logging and sampling facility by the drillers, where it was laid out on steel sawhorses/trestles or tables.

Troilus personnel then align and rough log the drill core where meterage is reviewed and recorded for core recovery and Rock Quality Designation (RQD). In general, core recovery is high (> 95%) with little core loss. Drill core is moved to the core logging tables (Figure 10-4) where Troilus geologists log lithology, veins, mineralization, texture, veins, and faults/fractures directly on computer to the Geotic database. All drill logs are vetted by Troilus managers before being finalized in the Geotic database. Drill core is marked using grease pencils where: red – sample interval, orange lithology contact, yellow – mineralization and white – alteration.

The Troilus geology personnel maintains a diamond drill core reference suite, or witness samples, of the main lithological units and alteration products on the property in order to maintain consistency in lithology nomenclature.

The core was then marked up for sampling in one or two-metre intervals. Earlier 2018 drill holes were broken up into more varied lengths. Sample tags are placed in the core box at the base of the sample interval and stapled to stay in the box.

Prior to sampling, all core is photographed wet and dry as part of the standard logging procedure. A special frame with white cover and lights is used to for the camera to maintain consistency in the photographs (Figure 10-5). A whiteboard is used to label the drill hole number, from and to, and core box number in the photograph.

Figure 10-4: Core Logging Tables

Source: AGP (2020)

Figure 10-5: Core Photo Set-up; fan is used to dry core.

Source: AGP (2020)

10.2.3 Drill Core Sampling

The sampling facility is adjoined to the logging area and is accessed by a garage door inside the building. Troilus has three core saws: two for the NQ drill core and one for PQ drill core.

Once the drill core has been marked up for sampling, it is stationed next to the sampling room, in the same facility, where the drill core is split by core saw. One half core is placed in the sample bag, the other is returned to the core box. The sample bag contains a copy of the sample tag and is marked with the sample number on the bag in permanent black marker.

The sample bag is sealed by zip tie and then placed with other sample bags in a larger white rice bag. The rice bags hold approximately 10 samples. The rice bags are reviewed by Troilus personnel and marked with the sample numbers and client code before the rice bag is sealed by zip tie and orange flagging tape. Rice bags are placed in wood prefabricated crates (on palettes) and is covered with a plywood cover, screwed closed, and strapped. Once enough crates are filled (approximately 30 rice bags) the transport company, Groupe Transcol Inc. (Transcol), based in Chibougamau, is called in for pick up and transport directly to ALS Global in Sudbury.

The core saw is cleaned after each sample and the sampling room is cleaned every night. Core boxes of the sampled core are kept on temporary racks outside the sampling room for temporary storage until they are moved to the exterior core storage area. Here, the core boxes are tagged with aluminium tags with the drill hole number, from and to, and core box Number. The aluminium tag is stapled to the end of the core box. Drill core is stored on site in covered metal core racks outside the core logging facility.

10.3 Previous Drill Methods and Logging

In the earlier drilling programs on the Property, before 1990, AQ (27 mm) and BQ (36.5 mm) size core was used and, in the early 1990s, NQ (47.6 mm) drill core was used (Evans, 2019b).

From 1986 to 1996, all casings were left in the ground. From 1997 to 1999, all casings from "KN" holes drilled during that period and located in the Z87 Zone and J4 Zone areas were removed, while casings for other "KN" holes and all "TN" holes were left in place. Between 2000 and 2005, all casings for "KN" holes were removed after completion and those for "TN" holes were partly left in the ground.

From 1986 to 2002, acid dip tests and Tropari instruments were used systematically. In 2003, a Reflex Multishot digital survey started to be used. The collars of all holes drilled in the vicinity of the Troilus deposit were surveyed using the mine grid coordinate system. For exploration holes outside the mine area, cut line grid coordinates were converted to the mine grid system. The elevations for these holes was estimated from topographic maps.

Drill holes prior to 1990 were converted to the metric system and verified by Inmet prior to inserting them into the database.

10.3.1 Drill Core Logging

Drill core logging was done for major and minor lithologies, alteration type, and mineralization. Over the years, the lithological naming conventions evolved, generally from volcanic origins to more intrusive origins.

RQD measurements were systematically taken during the 1991 drilling campaign. In following drill programs, RQD was done only on a few holes selected on each section drilled. In 2005, RQD measurements were again systematically collected.

10.3.2 Drill Core Sampling

Since 1986, a consistent sample protocol was employed at Troilus prior to shipping samples for analysis.

From 1986 to 1997, drill core was split, with half of the core placed in wood boxes that were tagged with Dymo tape and the remaining half sent to the laboratory for assaying (Evans, 2019b). All core samples was marked, tagged, placed in plastic bags, sealed, and temporarily stored in the secure core shack. When sufficient samples were accumulated, they were shipped by truck to the assay laboratory.

Before 1990, sample lengths in the earlier programs were not constant and depended on mineralization and geology, such as dykes, contacts, etc. (Evans, 2019b). In the subsequent programs, it was found that the mineralization was very diffuse throughout the geological units and systematic 1 m sample intervals were taken, regardless of the geology, within known mineralized zones; and up to 2 m sample intervals in surrounding intrusive rocks. Drill core samples were split into two parts with a hydraulic splitter: one half of the core was sent for assay and the other half was put back in the core boxes for future reference, metallurgical work, or additional check assaying. Since the mineralization consisted essentially of disseminated pyrite and given that there was not a good correlation between pyrite abundance and gold grade, the logging geologists found it virtually impossible to visually estimate gold grades.

From 1999 to 2002, most of the Z87 diamond drill core samples were three metres in length and most of the J4 Zone samples were 2.5 m in length. For the 2002 J4 Zone drilling, the mine laboratory adjusted the protocol to a 2.5 m length. In 2004, all sample lengths were reduced to two metre lengths.

In 1999, a new sampling and metallic sieve-based assay protocol was introduced. This protocol included increasing the sample length to three metres and was applied to all samples located within mineralized zones. This was done systematically, without considering geological contacts or dikes. The sample length for samples located outside the mineralized zones was set at two metres. Starting in 1999, whole core was sent for assay and a 10 cm to 20 cm length of core was retained as a witness of the interval.

The drill core for holes drilled up to 1996 was stored outside in core racks at the Opemiska Mine site in the town of Chapais but are now destroyed. The more recent core (post-1997) is stored in racks and pallets at the Project site.

10.4 Geotechnical and Hydrological Drilling

There has been a total of four pit-slope investigative geotechnical drill campaigns since Troilus acquired the project in 2018. The programs were carried out by WSP and included Packer testing and geotechnical logging. During these programs, there were dedicated geotechnical drill holes that were planned by WSP, as well as exploration drill holes planned by Troilus, which were logged geotechnically by WSP personnel, in addition to work done by Troilus geologists.

In 2020, five dedicated geotechnical holes were drilled, totaling 2,160 m, which focussed on the pit walls of the J Zone and the Z87 Zone. In 2021, nine dedicated geotechnical holes were drilled, totaling 2,281 m, focussed on the pit walls of the J, Z87 and SW Zones. In 2023, five dedicated geotechnical drill holes were completed, totaling 1,602 m, which focussed on the pit walls of the J Zone and X22 Zone.

In 2023, two surface geotechnical campaigns took place, which investigated the ground conditions below planned future infrastructure. In total, 56 drill holes were drilled, that totalled 470 m.

10.5 Metallurgical Drilling

In 2019, four dedicated drill holes, totalling 945 m, were completed entirely for metallurgical testwork. All four of these drill holes were in the J Zone.

10.6 Summary of Drill Intercepts

10.6.1 Z87 Zone

Initial drilling in 2018 began at the Z87 Zone with the focus on mineralization at depth. A southern extension of the Z87 Zone was discovered in a later drill campaign in late 2019. The Z87 South Zone has now been incorporated into the Z87 Zone.

Initial drilling in 2018 began at the Z87 with the focus on mineralization at depth. In 2019, extensions both to the north and south of Z87 were discovered. The Z87 South Zone and Z87 North Zone were both later incorporated into Zone 87. From 2020 to 2023, drilling mainly focused on infilling previously unexplored ground between the three former zones, as well as upgrading resources in the southern portion of Z87. In 2022, the J-87 Connector was discovered, which is the zone between Z87 and J Zone. Drilling targeted Z87 hanging-wall mineralization and discovered a mineralized, D2 structure oblique to dominant mineralization at Troilus. The structure runs from the southern tip of the existing J4 pit, to the centre of the western wall of the existing Z87 pit.

Table 10-2 lists selected drill hole intercepts with significant values. Figure 10-6 shows a cross-section of the 87 Zone at 13925 N.

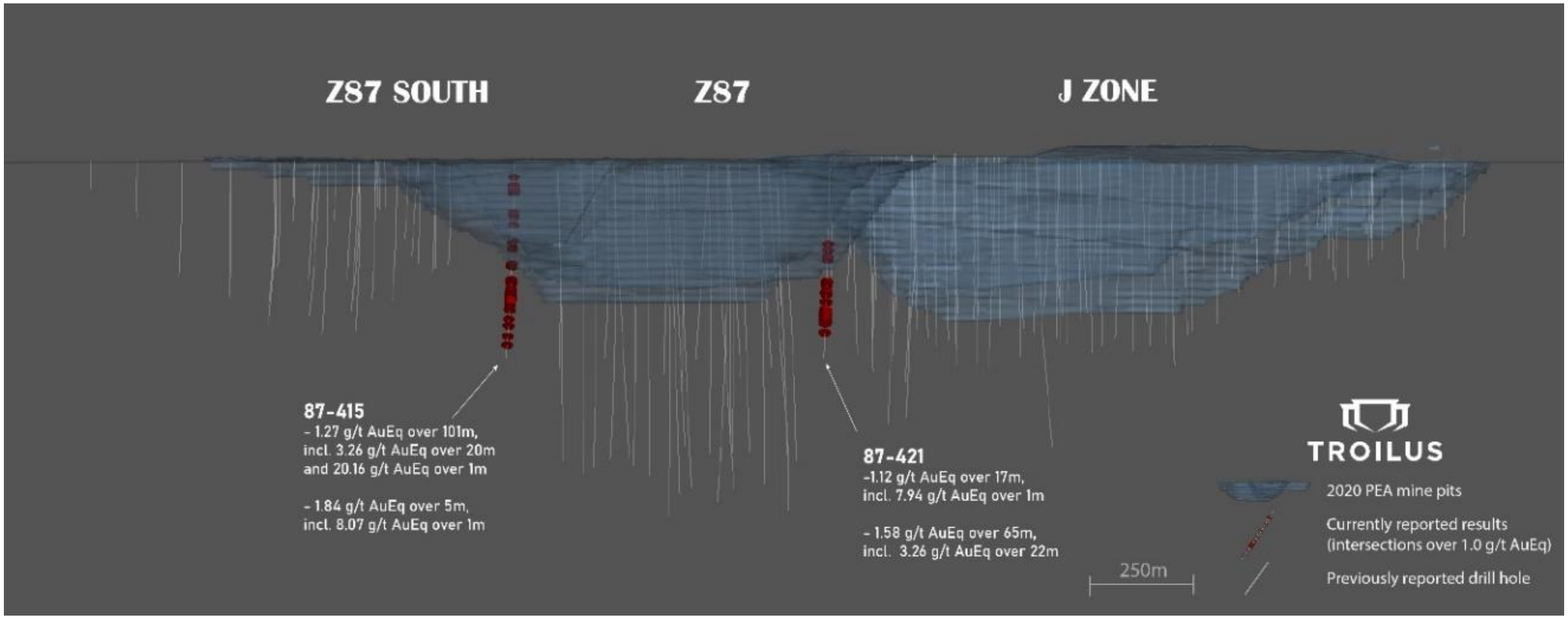
Table 10-2: Summary of Significant Drill Intercepts – Z87 Zone

DH No	Section		From (m)	To (m)	Width (m)	Au (gpt)	Cu (%)
TLG-Z8718-001	13650N		464	509	45	1.7	0.21
		including	472	477	5	6.09	0.54
TLG-Z8718-005	13750N		439	529	90	1.02	0.12
		including	458	464	6	1.57	0.25
		including	472	477	5	3.03	0.57
		including	520	528	8	2.36	0.11
TLG-Z8718-010	13600N		654	688	34	1.17	0.11
		including	660	666	6	1.88	0.08
		including	679	685	6	1.74	0.30
TLG-Z8718-017	13925N		625	632	7	0.61	0.09
			643	685	42	2.61	0.08
		including	671	673	2	42.30	0.12
			686	692	6	1.34	0.03
		including	686	688	2	3.02	0.02
TLG-Z8718-035	13875N		670	674	4	0.84	0.02
			689	770	81	1.44	0.13
		including	707	710	3	8.25	0.54
		including	751	753	2	2.77	0.37
		including	755	765	10	3.23	0.30
		including	767	769	2	2.91	0.04
			775	793	18	0.81	0.03
TLG-Z8718-044W	13925N		832	899	67	1.58	0.10
		including	874	876	2	10.03	0.35
		including	881	887	6	7.54	0.17
TLG-Z8718S-133	12800N		100	116	16	0.32	0.04
			214	282	68	0.86	0.03
		including	234	282	48	1.06	0.02
		including	270	276	6	5.02	0.02
TLG-Z8718S-136	12700N		177	183	6	1.35	0.03
			207	211	4	0.79	0.04
			223	243	20	0.43	0.11
		including	235	243	8	0.69	0.22

DH No	Section		From (m)	To (m)	Width (m)	Au (gpt)	Cu (%)
		including	239	241	2	1.80	0.27
			79	96	17	0.71	0.06
87-22-415	13275N	including	80	88	8	1.02	0.03
			151	161	10	0.88	0.02
			347	358	11	0.84	0.02
		including	347	348	1	5.07	0.02
			366	467	101	1.13	0.10
		including	406	426	20	3.00	0.22
		including	466	467	1	20.1	0.04
87-22-421	14050N		338	355	17	1.01	0.08
			377	386	9	0.82	0.10
			415	480	65	1.32	0.19
		including	431	453	22	2.77	0.34

Troilus Press Releases: 24 May 2018; 9 Jul 2018; 12 Sep 2018; 31 Oct 2018; 19 Aug 2019, 17 Aug 2022

Figure 10-6: Long Section of the Z87 Zone and J Zone; looking northwest



Source: Troilus Press Release 17 Aug 2022

Note: Z87 South Zone is now part of Z87 Zone

10.6.2 J Zone

In 2019, the drill program focussed on the extension of the mineralization at J Zone. The drill results confirmed that the mineralization agreed with previous drill campaigns. Troilus drillholes also demonstrated that mineralization continues to the northeast and to the southwest of the J Zone and at depth.

In 2020, the J4/J5 Zone were incorporated into what is now the J Zone. Drilling from 2020 to 2023 continues to grow mineralization along strike (north and south) and at depth. In 2021, mineralization was intersected to the west of the previously defined J Zone leading to approximately 150 m of mineralization expansion in this direction.

Table 10-3 lists selected drill hole intercepts in the J Zone with significant values. Figure 10-7 shows a selected cross-section of the J Zone at 14150N.

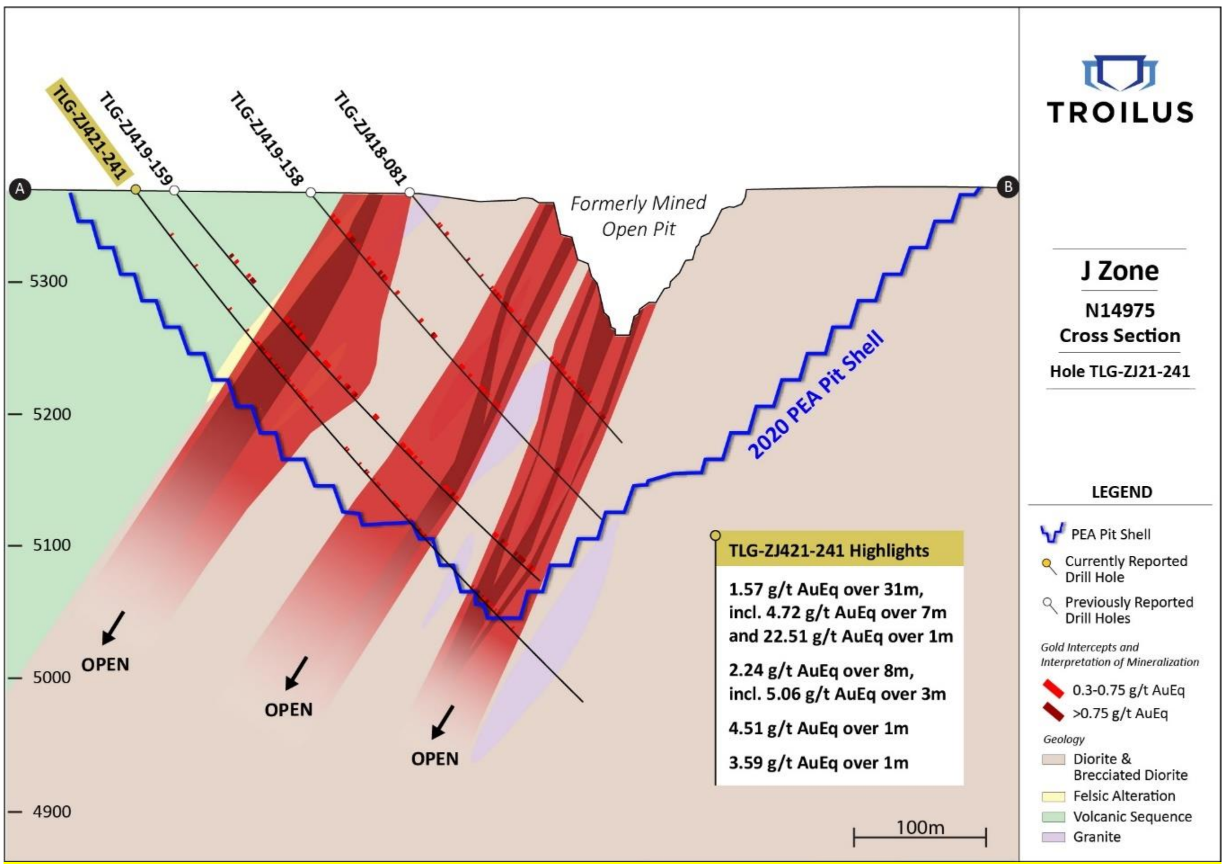
Table 10-3: Summary of Significant Drill Intercepts – J Zone

DH No	Section		From (m)	To (m)	Width (m)	Au (gpt)	Cu (%)
TLG-ZJ419-092	14150N		317	325	8.00	2.93	0.05
		including	317	319	2.00	9.61	0.10
			383	390	7.00	0.82	0.13
			397	406	9.00	1.96	0.08
		including	401	405	4.00	3.38	0.10
			422	441	19.00	0.95	0.10
		including	422	425	3.00	0.68	0.11
		including	427	433	6.00	1.06	0.10
		including	435	441	6.00	1.53	0.16
		including	439	440	1.00	5.22	0.64
TLG-ZJ21-226	14300N		93	161	68	0.71	0.27
		Including	103	112	9	0.9	0.47
		Including	118	128	10	1.08	0.39
		Including	151	159	8	1.08	0.42
TLG-ZJ21-235	14775N		102	104	2	1.54	0.06
			454	477	23	1.11	0.07
		Including	456	457	1	2.67	0.11
		Including	470	477	7	2.44	0.05
			507	510	3	1.67	0.03
TLG-ZJ21-241	14975N		146	177	31	1.5	0.05
		Including	150	157	7	4.63	0.05
		Including	150	151	1	22.4	0.04
			405	413	8	2.18	0.03
		Including	409	412	3	4.97	0.06
TLG-ZJ21-244	15075N		82	110	28	0.76	0.07
		Including	86.75	103	16.25	1.03	0.09

DH No	Section		From (m)	To (m)	Width (m)	Au (gpt)	Cu (%)
		Including	102	103	1	8.1	0.04
			280	311	31	2.04	0.04
		Including	299	311	12	4.35	0.04
		Including	309	310	1	27	0.03
TLG-ZJ21-251	15350N		138	154	16	1.63	0.05
		Including	148	153	5	4.07	0.06
		Including	148	149	1	14.65	0.05
			174	178	4	2.14	0.1
		Including	175	176	1	6.31	0.13

Troilus Press Releases: 26 Mar 2019, 12 May 2021, 8 Jun 2021, 7 Jul 2021, 21 Sep 2021

Figure 10-7: Cross Section 14975N – J Zone; looking northeast



Source: Troilus Press Release: 7 Jul 2021

10.6.3 X22 Zone

The X22 Zone is situated adjacent to the southwest of the Z87 Zone. Drilling was completed on the X22 Zone in 2022 and 2023 that included 76 drill holes, totaling 21,932 m. Zone X22 is hosted within a D2 structural corridor that overprints a tonalitic body within the Troilus diorite intrusion. Where D1 structures intersect this corridor, gold mineralization may occur.

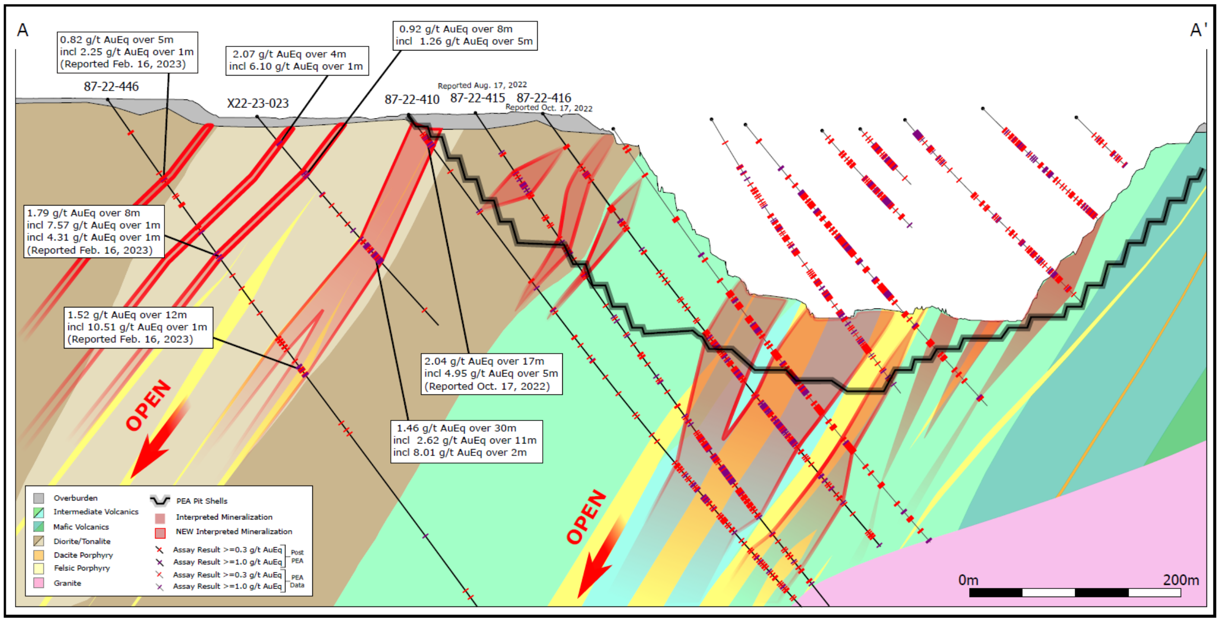
Table 10-4 lists selected drill hole intercepts in the X22 Zone with significant values. Figure 10-8 shows a cross section of the X22 Zone at 13275N.

Table 10-4: Summary of Significant Drill Intercepts – X22 Zone

DH No	Section		From (m)	To (m)	Width (m)	Au (gpt)	Cu (%)
X22-23-023	13275N		31	35	4	2.01	0.01
		Including	32	33	1	5.93	0.02
			151	181	30	1.43	0.02
		Including	170	181	11	2.54	0.05
X22-23-042	13075N		166	167	1	102.50	0.82
			287	369	82	0.70	0.10
		including	323	368	45	0.92	0.13
X22-23-074	12875N		215	226	11	0.87	0.10
		including	216.4	217.4	1	5.79	0.40
			277	308	31	0.72	0.08
		including	277	278	1	2.38	0.16
		including	286	287	1	1.52	0.35
		including	299	300	1	5.80	0.40
X22-23-071	12625N		256	258	2	4.36	0.08
		including	256	257	1	6.53	0.03
			309	389	80	1.32	0.30
		including	322	323	1	6.70	2.58
		including	379	389	10	7.63	1.51
X22-23-031	12475N		133	154	21	1.18	0.20
		including	142	148	6	2.04	0.37

Troilus Press Releases: 31 Mar 2023

Figure 10-8: Cross Section 13275N – X22 Zone; looking northeast



Source: Troilus Press Release: 7 Jul 2021

10.6.4 SW Zone

The SW Zone is situated approximately 2.5 km southwest of the Z87 Zone pit. In late 2019/ early 2020, the initial drilling of 8,500 m outlined a mineralized zone covering an area of 1.2 km x 0.5 m. From 2020 to 2023, Troilus completed more than 108,000 m of drilling in the SW Zone and expanded mineralization along strike, laterally and at depth. The SW Zone is now interpreted over an area of 2.5 km x 1.0 km.

Table 10-5 lists selected drill hole intercepts in the SW Zone with significant values.

Figure 10-9 shows a selected cross-section of the SW Zone at 9600 N.

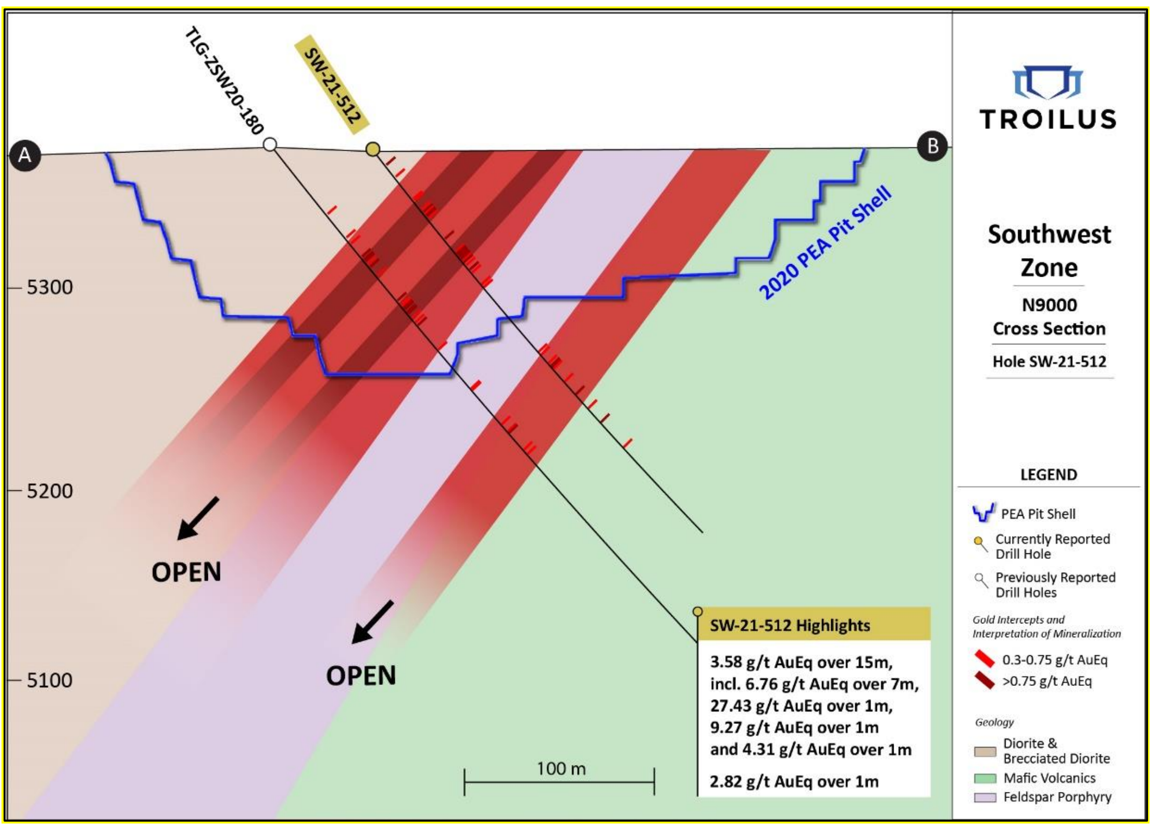
Table 10-5: Summary of Significant Drill Intercepts – SW Zone

DH No	Section		From (m)	To (m)	Width (m)	Au (gpt)	Cu (%)
TLG-ZSW20-203	9525N		439	442	3	6.54	0.077
		Including	439	440	1	17.8	0.078
			462	478	16	1.06	0.073
		Including	462	470	8	1.73	0.096
			485	506	21	1.04	0.041
		Including	488	489	1	2.52	0.064
		Including	497	498	1	2.08	0.011
		Including	504	505	1	11.15	0.117
TLG-ZSW20-204	9525N		59	66	7	1.08	0.004
		Including	61	64	3	1.6	0.002
			142	146	4	1.71	0.287
			315	324	9	1.23	0.201
		Including	315	320	5	1.8	0.304
			346	366	20	1.69	0.193
		Including	357	366	9	238	0.266
			574	575	1	3.19	0.212
			599	601	2	8.57	0.079
TLG-ZSW20-208	9700N		248	266	18	1.14	0.0169
		Including	250	257	7	2.33	0.0207
		Including	252	257	7	2.33	0.0246
		Including	265	266	1	1.38	0.01
TLG-ZSW20-214	10000N		193	208	15	0.93	0.052
		Including	196	197	1	3.07	0.016
		Including	204	207	3	1.76	0.052
SW-21-512	9030N		42	49	7	0.89	0.01
		Including	46	48	2	1.77	0.02
			71	86	15	3.51	0.04
		Including	72	79	7	6.7	0.04
		Including	73	74	1	27.4	0.01
		Including	78	79	1	9.22	0.03
		Including	83	84	1	4.23	0.06
SW-21-537	9075N		59	78	19	1.08	0.03
		Including	69	74	5	3.12	0.02
			261	268	7	1.16	0.02
			316	322	6	1.11	0.03
		Including	319	320	1	5.26	0.02
SW-22-360	10000N		11	26	15	3.06	0.01
		Including	13	18	5	8.25	0.02

DH No	Section		From (m)	To (m)	Width (m)	Au (gpt)	Cu (%)
			211	231	20	0.8	0.02
		Including	214	221	7	1.48	0.03
			240	243	3	1.18	0.01
		Including	241	242	1	2.88	0.01
			259	262	3	1.65	0.01
		Including	261	262	1	3.94	0.01
SW-22-616	9150N		2.73	9	6.27	1.26	0.01
		Including	5	6	1	2.88	0.01
			78	94	16	0.69	0.05
		Including	78	79	1	2.93	0.06
		Including	87	88	1	3.71	0.06
			427	437	10	1.35	0.03
		Including	427	428.8	1.8	2.95	0.02

Troilus Press Releases: 12 Jan 2021; 9 Feb 2021; 24 Feb 2021; 16 Mar 2021; 17 Aug 2021; 20 Jan 2022; 21 Apr 2022; 4 May 2022.

Figure 10-9: Cross Section 9600 N – SW Zone; looking northeast



Source: Troilus Press Release: 17 Aug 2021

10.7 Exploration Drilling

10.7.1 Allongé Zone

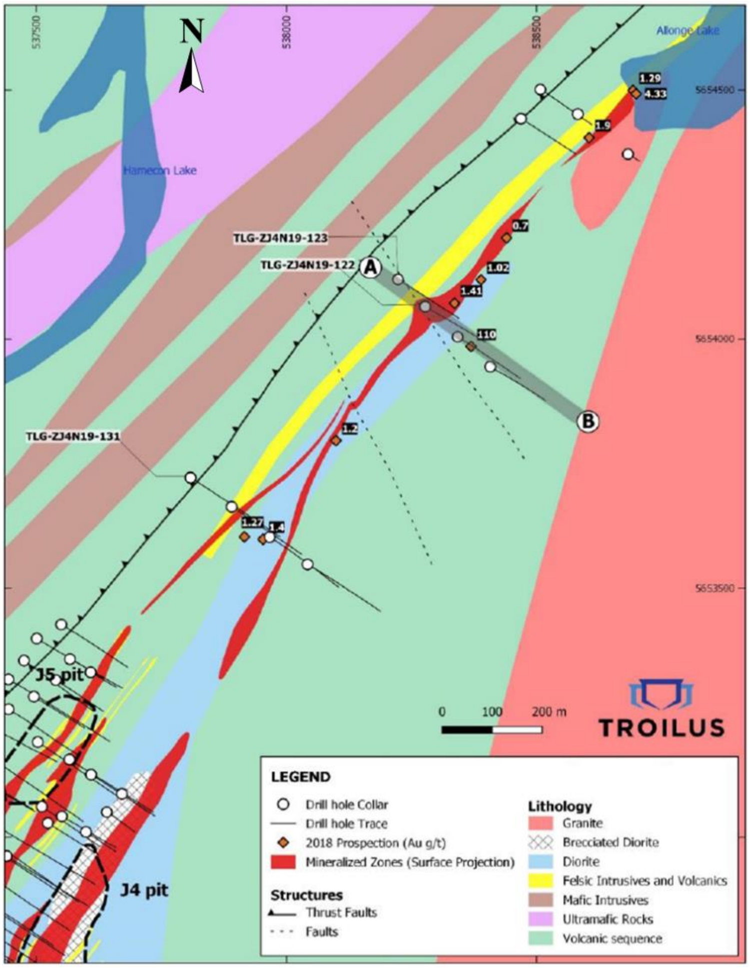
To follow up on results of surface grab samples and a single historic drill hole (KN-684), Troilus completed 12 drill holes, totalling 2,193 m, in the Allongé (previously J4N Zone) Target along three fences. This zone is situated approximately 350 m to 1400 m northeast of the J Zone. Six of the drill holes had intersections, between 2 m and 12 m of greater 0.3 gpt Au. The most significant intersections found in the Troilus drilling, approximately 900 m northeast of the J Zone (Section 16525N), and roughly 100 m northeast of the historic KN-684 drill hole. These are positive indications of gold mineralization and warrant further investigation.

Table 10-6 summarizes the significant intersections in the Allonge Target. Figure 10-10 shows the location of the Allongé Target drilling.

Table 10-6: Summary of Significant Drill Intercepts – Allongé Zone

DH No	Section		From (m)	To (m)	Width (m)	Au (gpt)	Cu (%)
TLG-ZJ4N19-122	61525N		26	48	38	0.47	0.14
		including	44	48	4	1.05	0.31
TLG-ZJ4N19-123	61525N		71	85	14	0.57	0.01
			97	105	8	0.23	0.06
			111	119	8	1.03	0.14
		including	113	115	2	2.50	0.17

Figure 10-10: Plan View – J4N or Allongé Zone



Source: Troilus (2019)

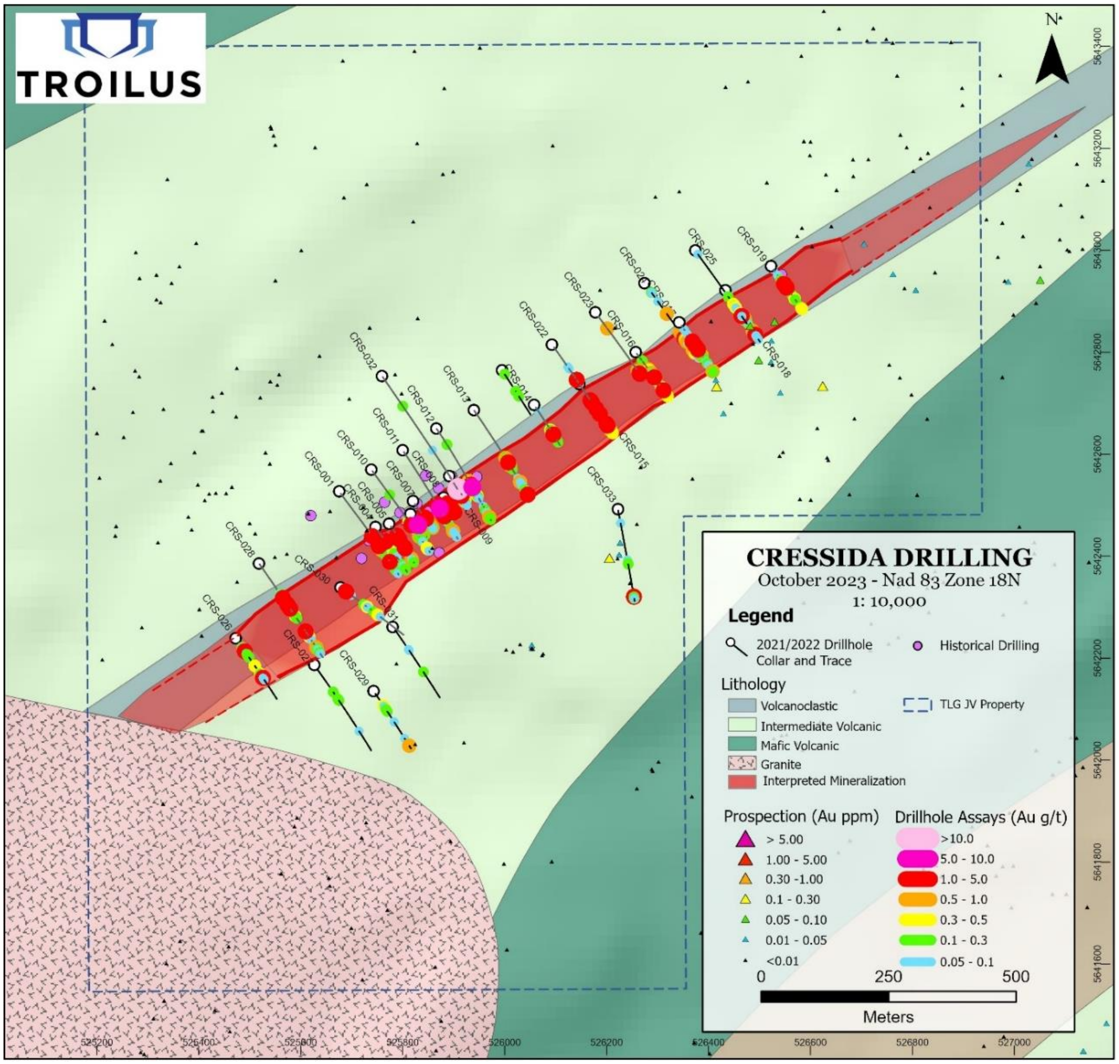
10.7.2 Cressida Target

In December 2019 and March 2020, Urban Gold carried out a four-hole drill program totalling 689 m, targeting the previously delineated ore zone. The highest results returned values of 1.02 g/t Au over 5.6 m and 0.9 g/t Au over 17.55 m, showing an economic potential of the deposit and extending the strike length to the northeast.

In 2021, soil sampling and prospecting done by Troilus across the claim block discovered anomalous gold values both 500 m to the northeast and 250 m to the southeast of the main zone.

In 2021 and 2022, a 6,070 m drill program (31 drillholes) was carried out in two phases targeting geophysical anomalies. In late 2021, the Phase 1 program totalled 4,676 m over 23 drillholes, targeting the previously identified ore zone, extending it to roughly 200 m vertical depth and approximately 950 m along the strike length. In the summer of 2022, the Phase 2 drill program targeted a highly conductive IP anomaly southeast of and parallel to the main zone. Two drill holes also tested the known mineralized trend further to the southwest and at depth. (Figure 10-11). Table 10-7 summarizes some of the highlights of the Phase 1 and Phase 2 drill campaigns.

Figure 10-11: Troilus Frotêt Project – Cressida Target



Source: Troilus (2023)

Table 10-7: Summary of Drill Results – Cressida Target

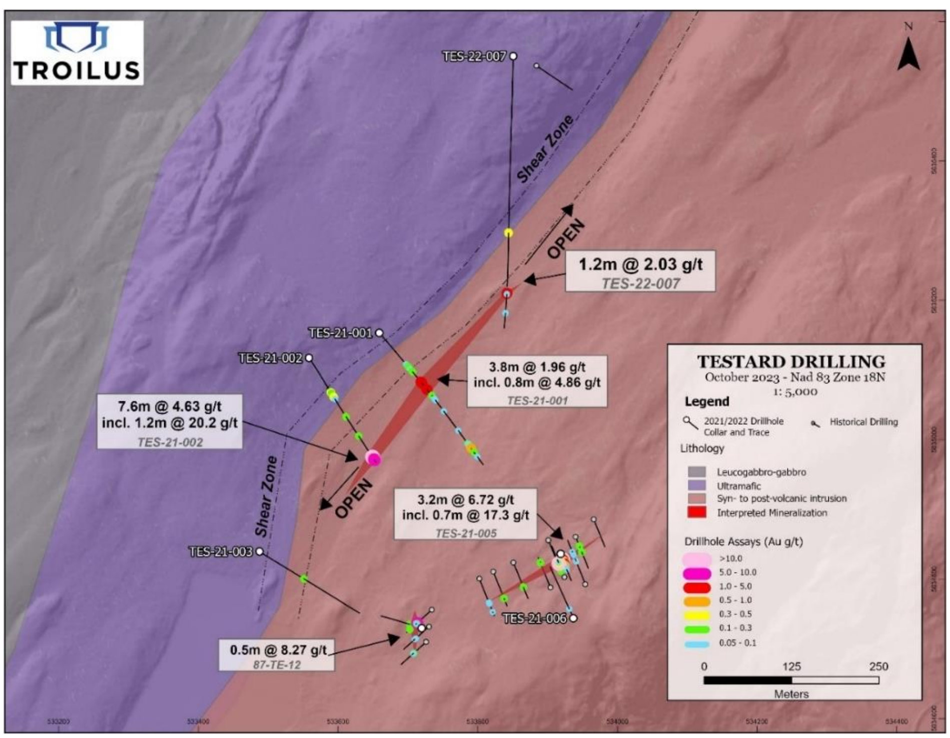
Drill Hole	From (m)	To (m)	Interval (m)	Au (g/t)
CRS-21-006	28	44	16	1.64
including	38	41	3	6.23
CRS-21-011	187	202	15	1.23
including	201	202	1	8.16
CRS-21-012	179	203	24	0.759
including	202	203	1	9.45
CRS-21-008	21	33	12	0.88
CRS-21-023	245	263	18	0.615
CRS-22-032	380	387	7	10.68
including	382	383	1	66.8

Note: drill intervals reported are down hole core lengths as true thicknesses cannot be determined with available information

10.7.3 Testard Target

In 2021, a drill hole program of six drill holes, totalling 1,280 m, was completed to target different structures in the area that had the potential to carry gold mineralization, including testing extensions of the higher-grade mineralization below surface at the main showing (Figure 10-12).

Figure 10-12: Troilus Frotêt Project – Testard Target; highlighting drilling results



Source: Troilus (2023)

Holes TES-21-001 and TES-21-002 intersected previously unknown quartz veins with high-grade gold and silver values within a strongly deformed and altered tonalite approximately 400 m northwest of the main Testard outcrop. Hole TES-22-007 is interpreted to have intersected the same structure as drill hole -001 and -002. This appears to extend the mineralized zone approximately 170 m along strike to the northeast and to an estimated vertical depth of roughly 420 m.

Best intervals in hole TES-21-002 are 4.63 g/t Au and 20.36 g/t Ag over 7.6 m, including 20.2 g/t Au and 76.9 g/t Ag over 1.2 m, and 7.12 g/t Au and 68.45 g/t Ag over 1.4 m. Hole TES-21-001 intercept highlights include 1.96 g/t Au and 19.12 g/t Ag over 3.8 m, including 2.68 g/t Au and 30.48 g/t Ag over 1.8 m and 4.86 g/t Au, 38.8 g/t Ag over 0.75 m.

Drillhole TES-21-005 targeted and intersected mineralized gold bearing structures at depth below the main showing at the contact between a tonalite and a strongly sheared mafic-ultramafic dyke. Intercept highlights include 6.72 g/t Au and 26.71 g/t Ag over 3.2 m, including 17.3 g/t Au and 75.3 g/t Ag over 0.7 m.

Table 10-8 summarizes best intercepts in contact with an ultramafic sill.

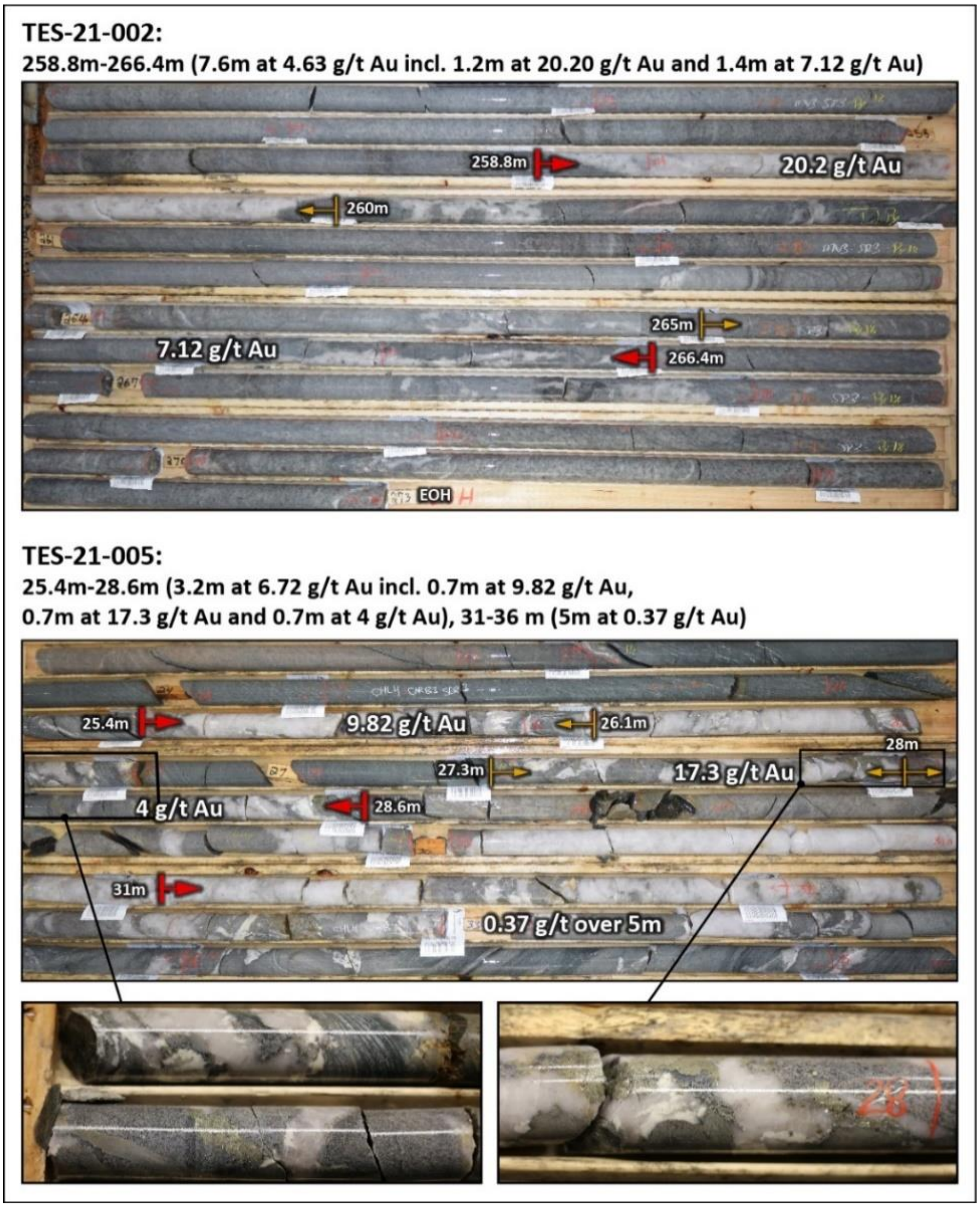
Table 10-8: Summary of Drill Results – Testard Target

Drill Hole	From (m)	To (m)	Interval (m)	Au (g/t)
TES-21-001	146.0	149.8	3.8	1.96
including	148.0	149.8	1.8	2.68
	167.3	168.0	0.8	4.86
	326.0	327.0	1.0	0.61
TES-21-002	103.6	105.0	1.5	0.50
	258.8	266.4	7.6	4.63
including	258.8	260.0	1.2	20.20
including	265.0	266.4	1.4	7.12
TES-21-005	25.4	28.6	3.2	6.72
including	25.4	26.1	0.7	9.82
including	27.3	28.0	0.7	17.30
including	28.0	28.6	0.7	4.00
	31.0	36.0	5.0	0.37
	50.0	51.0	1.0	1.02

Note: Drill intervals reported are down hole core lengths as true thicknesses cannot be determined with available information

Figure 10-13 illustrates the mineralized structures encountered consist of shear-hosted quartz-tourmaline-carbonate-albite veins contained within a sericite-silica-carbonate ± hematite and chlorite altered tonalite and chlorite-carbonate altered mafic to ultramafic dykes. Veinlets and patches of specular hematite appears to be part of a distal alteration halo within the tonalite. The best gold and silver values were obtained from veins that contain disseminated, up to 20% pyrite, with locally trace chalcopyrite, malachite and molybdenite. Different vein textures have been observed in core including laminated, extensional, and breccia-type veins. Further drilling is required to better constrain the azimuth and dip of the different mineralized trends that were intercepted.

Figure 10-13: Troilus Frotêt Project – Testard Target; core photos of TES-21-002 and TES-21-005



Source: Troilus (2022)

10.7.4 Pallador Target

In 2022, five drill holes were completed on the Pallador Target, totaling 2,240 meters to test the geology below the glacial cover and target interpreted magnetic features proximal to the up-ice origin of the mineralized boulder fields. The highest results returned values up to 2.45 g/t Au over 1 m and

4.43 g/t Au over 1 m from the same drill hole (RCK-22-004). Mineralization was associated with sheared and silicified gabbro containing intermittent quartz veining and up to 5% pyrite locally.

In September 2023, two drill holes were completed, totalling 653 m, to test chargeability anomalies at the Rocket showing. Drilling intersected the same highly magnetic gabbro encountered in the 2022 drilling at approximately 1 km along strike to the northwest. The best result from this drilling returned 2.93 g/t Au over 1 m from a locally sheared gabbro with disseminated and vein-hosted pyrite.

10.8 AGP Opinion

AGP considers the drilling was undertaken in accordance with industry standards and best practices without any major adverse aspects that could have materially impacted the accuracy and reliability of the resource estimate.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Pre-2018

11.1.1 Analytical Laboratories

Prior to 1997, samples were shipped off site to certified assay laboratories. During mining operations, from 1997 to 2007, samples were assayed on-site.

During the first drilling programs (1986 to 1991), several independent laboratories, including Swastika Laboratories (Swastika), based in Swastika, Ontario, were used for assaying the core samples. Bondar-Clegg and Chimitec (now part of ALS) were also used.

Following an extensive assaying comparison program in 1992 between several laboratories using different techniques, Swastika was retained to do most of the analyses from 1992 to 1997.

From 1997 to 2007, when Troilus was in operation, Inmet used their own laboratory set up at the mine. The mine laboratory was equipped with modern state-of-the-art equipment and staffed with highly qualified personnel.

11.1.2 Sample Preparation and Analysis

Before 1992, Bondar-Clegg and Chimitec used a half assay-ton fire assay technique with a direct coupling plasma (DCP) finish. At Swastika, it was determined that the one-assay tonne fire assay with gravimetric finish technique used by Swastika was more accurate for assaying gold than the half assay ton method used at the other laboratories. Consequently, from 1992 to 1999, all samples were assayed for gold by one-assay tonne fire assay with a gravimetric or AA finish depending on the size of the "doré bead". If the bead was visually judged too small to be weighed, then the bead was dissolved, and an AA finish was used. Copper and silver were analyzed by AA spectrometry.

Prior to assaying, the original one metre split core sample, weighing approximately 2.7 kg, was entirely crushed down to 0.25 in. Then, 350 g was pulverized to –150 mesh (105 microns) and a one-assay ton (29.17 g) fire assay was done. The rest of the sample (pulp and reject) was stored for future use.

In 1999, a new sampling and metallic sieve-based assay protocol was introduced following the studies and recommendations by Pitard (1999) (Pitard protocol) and included increasing the sample length to three metres and was applied to all samples located within mineralized zones. The Pitard protocol involved assaying a much larger sample than that used for the standard fire assay in the previous programs (1,000 g versus 30 g). This protocol was designed to reduce the Fundamental Error (i.e., error generated by sample and subsample weights), the Grouping and Segregation Error (i.e., error generated by gold segregation and the way samples and subsamples are split), the Extraction Error (i.e., error generated by poor sample recovery), and the Preparation Error (i.e., error generated by excessive loss of fines). The Pitard Protocol for assaying Troilus diamond drill core involved:

- crush the entire three metre NQ core sample (14 kg) down to 16 mesh (0.04 in.)
- split a one-kilogram sample using a rotary divider
- pulverize the entire one-kilogram sample for no longer than 90 seconds to minimize smearing

- screen the entire one-kilogram sample using a 150-mesh screen
- perform as many one-AT fire assay on the +150-mesh fraction as needed to assay the whole +150 fraction
- perform two one-AT fire assays on the –150 mesh fraction
- the final assay value is the weighted average of the results from both fractions

Starting in 2004, the Pitard Protocol for diamond drill core was adjusted to two metre core length (ten kilograms). The rest of the procedure remained the same. Assay data compilation from the 2004 and 2005 diamond drilling programs showed that reducing the sampling length to two metres did not increase the sampling error significantly.

11.2 Troilus (2018 – 2020)

11.2.1 Analytical Laboratories

For the drilling completed in 2018, samples were sent to the following independent certified assay laboratories, AGAT Laboratories Ltd. (AGAT), based in Mississauga, Ontario; and ALS Ltd. (ALS), based in Sudbury, Ontario. For drilling completed in 2019 and 2020, all samples were sent to ALS in Sudbury.

Both labs, AGAT and ALS, have been assessed by the Standards Council of Canada (SCC), and conform to the requirements of ISO/IEC 17025:2005 General Requirements for the Competence of Testing and Calibration Laboratories standard; and ISO 9001:2015. The labs are recognized as an Accredited Testing Laboratory for a number of specific tests, including gold fire assaying, which are listed on the SCC website (www.scc.ca).

11.2.2 Sample Preparation and Analysis

In 2018, Troilus had their samples prepared and analyzed by AGAT and by ALS. From December 2018, Troilus only used ALS for sample preparation and analysis.

At AGAT and ALS, all samples were weighed prior to preparation and all samples were prepared by crushing the sample to 85% passing 75 microns on 500 g splits. Samples sent to ALS were prepared at their laboratory in Sudbury and the analysis was completed at the laboratory in Vancouver.

At AGAT, samples were assayed for gold by fire assay (AGAT Code: 202-552) with a 50 g charge with an Induced Coupled Plasma – Optical Emission Spectroscopy (ICP-OES) finish. Sample results greater than 3.5 ppm Au were re-analyzed with a gravimetric finish. This was changed to an Atomic Absorption (AA) finish in May 2018. A multi-element analysis was used for 23 elements (AGAT Code: 201-079). Samples underwent a sodium peroxide fusion followed by ICP-OES finish. Copper was analyzed as part of the multi-element suite; however, silver was not included.

At ALS, samples were assayed for gold by fire assay (ALS Code: Au-AA24) with a 50 g charge with an AA finish. Sample results greater than 3.5 ppm Au were re-analyzed with a gravimetric finish (ALS Code: Au-GRA22). A multi-element analysis was used for 33 elements (ALS Code: ME-ICP61). Samples underwent a four-acid digestion followed by Induced Coupled Plasma – Optical Atomic Spectroscopy (ICP-AES) finish. Copper and silver were analyzed as part of the multi-element suite.

In December 2018, Troilus retained an external consultant, Jack Stanley of Analytical Laboratory Consultant Ltd., to carry out an audit of both laboratories, who concluded that both facilities were following industry standards.

For the 2019 – 2020 drill programs, all samples were sent to ALS in Sudbury for preparation and for specific gravity measurements. Prepared samples were forwarded to ALS in Vancouver for analysis.

In February 2019, Troilus requested specific gravity to be measured by ALS (Sudbury) (ALS Code: OA-GRA08).

In May 2019, a decision was made to use two metres of split NQ core and apply the metallic sieve gold assaying protocol for all core samples. A fine crushing to 70% less than 2 mm was performed. The sample was divided so that 1.2 kg to 1.5 kg was used for analysis. The sample of 1.2 kg to 1.5 kg was then pulverized to 95% passing 106 mesh. Approximately 50 g was recovered for ME-ICP61 analysis of 33 elements by four acids inductively coupled plasma atomic emission spectroscopy (ICP-AES). The remainder of the sample was screened to divide the fraction larger and smaller than 106 mesh. The portion smaller than 106 mesh was analyzed in 50 g by fire assay. The portion larger than 106 mesh was fully analyzed. The values were then combined by weighted calculation. Both results were transmitted to Troilus by a certificate certified by the laboratory.

11.3 Density Determinations

11.3.1 Z87 Zone, J Zone, X22 Zone

Between 2019 and 2023, Troilus collected density measurements from core samples throughout the Z87, J and X22 Zone. Density measurements were carried out by ALS (Sudbury) (ALS Code: OA-GRA08) on samples sent for assay analysis using water immersion (wet/dry) method.

A total of 132,983 measurements were collected from 384 drill holes, across all three zones and were found to be lithologically controlled, with little variation in lithological densities between the zone areas. Densities were assigned by mean density by lithology. Overburden was assigned a density of 2.2.

Table 11-1 presents the descriptive statistics for the Z87, J an X22 Zones by lithology.

Table 11-1: Descriptive Statistics for Density by Lithology – Z87 Zone, J Zone, X22 Zone, 2018 – 2023

Lithology	Code	Count	Min	Max	Mean	Median	StDev	CV
FP	61	16386	1.97	3.50	2.73	2.71	0.08	0.03
IFP	62	114	2.64	3.04	2.82	2.75	0.11	0.04
Bas Andesite	63	15597	2.19	3.47	2.80	2.79	0.08	0.03
Mag Breccia	64	in SW Zone only			2.87			
Tonalite	65	11619	2.24	3.61	2.72	2.72	0.05	0.02
I2J	66	22252	2.02	3.79	2.79	2.79	0.05	0.02
V2	67	61833	2.06	3.67	2.76	2.75	0.06	0.02
V3	68	996	2.61	3.13	2.93	2.94	0.09	0.03
V3T	69	1074	2.22	3.25	2.87	2.88	0.11	0.04
I1B Parker	70	967	2.53	3.11	2.72	2.65	0.14	0.05
I1B Dykes	71	2145	2.07	3.42	2.64	2.62	0.06	0.02
Overburden	9				2.20			

Note: StDev – Standard Deviation; CV –coefficient of variation

11.3.2 SW Zone (2019-2023)

During the 2019-2023 drilling campaigns, Troilus collected density readings collected for all sample intervals. Density measurements were carried out by ALS (Sudbury) (ALS Code: OA-GRA08) on samples sent for assay analysis using water immersion (wet/dry) method.

A total of 112,878 density measurements were collected by Troilus from drill core during the 2019 - 2023 drill programs in the SW Zone. The density assignment for the SW Zone is based on the mean density values within each lithology. Density for Overburden was assigned the value of 2.20.

Table 11-2 presents the statistics for density in the SW Zone by lithology.

Table 11-2: Descriptive Statistics for Density by Lithology – SW Zone, 2019 – 2023

Lithology	Code	Count	Min	Max	Mean	Median	StDev	CV
FP	61	2.14	3.41	2.72	2.7	0.08	0.03	2.14
IFP	62	2.15	4.63	2.76	2.74	0.09	0.03	2.15
Bas Andesite	63	2.51	3.12	2.75	2.75	0.06	0.02	2.51
Mag Breccia	64	2.08	3.59	2.87	2.87	0.10	0.03	2.08
Tonalite	65	not in SW Zone						
I2J	66	2.26	3.71	2.80	2.8	0.06	0.02	2.26
V2	67	2.43	3.26	2.75	2.75	0.07	0.02	2.43
V3	68	1.81	3.93	2.92	2.94	0.10	0.03	1.81
V3T	69	2.54	3.85	2.82	2.81	0.10	0.03	2.54
I1B Parker	70	2.43	3.24	2.76	2.75	0.12	0.04	2.43
I1B Dykes	71	not in SW Zone						
Overburden	9				2.20			

Note: StDev – Standard Deviation; CV –coefficient of variation

11.4 Quality Assurance/Quality Control (QA/QC)

Troilus follows their internal Quality Assurance and Quality Control (QA/QC) procedures to assess drilling results. Troilus maintains written Standard Operating Procedures that lay out the protocols. The protocol used for insertions of these samples were as follows:

- blank (1 in every 25 samples)
- duplicates (1 in every 25 samples)
- standards (CRM) (1 in every 25 samples)

Analytical QA/QC failures are identified as:

- any blank sample that reported >0.1 g/t Au
- any CRM result that reported with a difference >3 standard deviations from the certified mean or recommended value for the standard
- more than two sequential CRM results that reported with differences >2 standard deviations from the certified mean or recommended value, having the same positive or negative bias

Results were tracked as part of the standard QA/QC procedures. Failures were investigated and samples were re-assayed as required.

Blanks

Coarse blank materials were inserted into the sample stream at a rate of one each for every 25 samples for all drill programs. The material for the blanks came from the Parker Lake Granite, situated to the southeast of the mineralized zones. For the 2018 drilling, Troilus employed the granite material from the end of drill holes; or broken rock coming from an outcrop located well inside the Parker Lake Granite. For the 2019 and 2020 drilling, Troilus used exclusively coarse material from the Parker Lake granite outcrop.

Certified Standards

Troilus has used 14 commercially produced Certified (or Standard) Reference Materials (CRMs) throughout the drill programs since 2018. The CRMs are sourced from Ore Research & Exploration PL, based in Perth, Australia.

Table 11-3 summarized the CRMs with their ‘recommended values’.

Table 11-3: Standard Reference Materials (SRMs) and Recommended Values

Troilus Number	SRM	Source	Au (gpt)	Cu (ppm)	Ag (gpt)	Year
S1	OREAS 209	Ore Research & Exploration PL	1.580	76	0.264	2018 - 2021
S2	OREAS 215		3.540	-	-	2018 - 2021
S3	OREAS 217		0.338	-	-	2018 - 2021
S4	OREAS 92		-	2294	0.700	2018 - 2019
S5	OREAS 922		-	2122	0.888	2018 - 2021
S6	OREAS 239		3.550	-	0.244	2021
S7	OREAS 235		1.590	-	0.135	2021
S8	OREAS 231		0.542	-	0.177	2021
S9	OREAS 153b		0.313	6780	1.400	2021 - 2023
S10	OREAS 254b		2.530	-	0.453	2021 - 2023
S11	OREAS 605b		1.720	50300	1015.000	2021 - 2022
S12	OREAS 620		0.685	1730	38.500	2021 - 2022
S13	OREAS 905		0.100	6380	-	2021 - 2022
S14	OREAS 506		0.364	4440	1.800	2021 - 2023

The CRMs were chosen to represent different grade ranges for gold and copper on the Project. All the CRMs are individually packaged in 30 g packets and were inserted with the drill core samples with sequential sample tags at a rate of one for every 25 samples.

The results were plotted by Troilus in chronological order on graphs depicting the 'recommended value' as well as plus/minus two and three times the standard deviation of the dataset to provide a check of the precision of the assays.

Duplicates

Duplicates were collected through out all drilling programs since 2018. Only between July 2019 and July 2020 collection of duplicates was paused. In mid-2020, Troilus re-established the collection of duplicate sample data on the SW Zone drill program and all succeeding drill programs. The duplicate samples were conducted on the pulps and rejects returned to Troilus post analysis. The samples were nominally selected based on mineralized domains and the pulps and rejects were sent to either ALS or SGS for analysis.

11.4.1 QA/QC (pre-2018)

The following is taken from RPA (2019b):

> *Several laboratories and assay methods were used in the course of the different drilling programs, and a number of re-assay and check assay programs were carried out over the years. Also, several studies on the heterogeneity and/or nugget effect of gold were carried out and are listed in Boily et al. (2008). From 1997 onward, Inmet operated an internal assay laboratory where gold and copper grades were reconciled with head grades from the operating mill (RPA, 2019b).*
>
> *Prior to 1999, during the assaying process, each laboratory did a systematic check assay every 10 to 15 samples. All samples assaying more than 1.0 g/t Au were re-assayed from a second pulp and all those assaying greater than 2.0 g/t Au were*

assayed a second time from the rejects. All assay laboratories routinely inserted in-house reference materials and certified standards.

Since 1993, Inmet used in-house reference materials, CANMET Mining and Mineral Sciences Laboratories (Department of Natural Resources Canada) (CANMET), CRMs and blanks in each shipment to the assay laboratories. Over 20 different in-house reference materials and CRMs were used by Inmet over time. All these in-house control samples were first pulverized and bagged (35 g) and then inserted after every 50 samples using the same sequential numbers as the core samples. After approximately every 10 control samples, a CANMET CRM or a blank was inserted instead of the in-house control sample.

Results from quality control programs (reference samples, CRMs, re-assays, and duplicate assays) are used to qualify reliable assay data. There are no data on the standards used by the off-site laboratories prior to 1993 and/or the results of their quality control. However, no major problems were reported in the assays from the drilling programs and differences between the original values and the second assays and/or duplicates were judged to be acceptable.

In a report dated March 1994, the Coopers & Lybrand Consulting Group compiled the different studies on the accuracy and precision of the assays carried out by Inmet and concluded that the relative accuracy for the gold grade at Troilus is ±15%. After 1994, a number of tests and studies on the heterogeneity of gold at Troilus were carried out for Inmet by various consulting firms. Pitard (1999) reviewed this work and concluded that a target of ±15% variance in the gold assay results was achievable and that a sampling protocol modification was required to reduce sampling error to this level.

In late 1998 and early 1999, approximately 1,427 m of core from the mineralized zones from 12 holes were re-sampled and assayed in two separate programs. Independent laboratories used for the assaying included SGS Lakefield Research Ltd. (SGS) and the Centre de Recherche Minérale. This program was designed to compare the newly introduced 1,000 g screen metallic sampling and assays (Pitard Protocol) with the historical 30 g sampling assay protocol. From this program, Inmet concluded that the relative difference between the two data sets was less than 2% and that there was no overall bias between the two protocols. It was concluded that the 1,000 g screen metallic protocol reduced the sampling error and therefore provided a much better estimate of the gold contained in any given sample and improved the ability to estimate grades locally. This protocol was adopted as the sampling protocol going forward.

In 1997, external check assays at Swastika Laboratories (Swastika), based in Swastika, Ontario, and Chimitec (now part of ALS) indicated that the Troilus laboratory was underestimating gold values by approximately 10% to 15%. The Swastika and Chimitec assays were within 5%. The 1997 drilling program targeted Z87 close to the pit limits.

Following the introduction of a new sampling and assay protocol in 1999 (Pitard Protocol), modifications were made to their quality control procedures. In addition to the insertion of in-house reference material and/or CRMs, approximately 10% of all the samples assayed were randomly selected and their rejects sent back to the laboratory to be re-assayed using the same assay protocol (duplicates).

An internal Inmet report (Boily, 2005), based on external check assays and the mine laboratory gold reference standards, concluded that the Troilus laboratory assays were not biased.

11.4.2 QA/QC (2018-2019)

The QA/QC program included blank materials and CRMs. Four CRMs were used during all drill programs on the Property. A fifth CRM (S4) was only used in the initial seven drill holes of 2018.

Table 11-4 shows a summary of the QA/QC samples submitted during the 2018 and 2019 drilling program on the Z87 Zone and J4/J5 Zone.

Table 11-4: Summary of Troilus QA/QC Program, 2018 – 2019

Description	2018 Number of Samples (% of database)	2019 Number of Samples (% of database)
Total Number of Samples	28,334	18,729
Number of Control Samples	6,449 (22.8%)	2,492 (13.3%)
Distribution		
Blanks	1,294 (4.6%)	829 (4.4%)
Blanks (BP)	383	829
Blanks (other)	912	-
Lab Duplicates	3,708 (13.1%)	815 (4.4%)
CRM samples	1,447 (5.1%)	848 (4.5%)
OREAS 209 (S1)	283	200
OREAS 215 (S2)	329	207
OREAS 217 (S3)	340	239
OREAS 92 (S4) *	32	-
OREAS 922 (S5)	463	202

* OREAS 92 was used for the initial seven drill holes of 2018

Blanks

For the 2018 drilling, the Parker Lake Granite material used for blanks was taken from the ends of selected drillholes, outcrop and in a few instances from silica sand from nearby Lac a la Croix (BSS). The drill holes ends were labeled:

- BP Parker Lake Granite outcrop
- B1 TLG-Z8718-002
- B2 TLG-Z8718-009

- B3 TLG-Z8718-010
- B4 TLG-Z8718-011
- B5 TLG-Z8718-020
- B6 TLG-Z8718-037
- B7 TLG-Z8718-049
- BSS silica sand (Lac à la Croix)

Results from the blanks found 11 failures out of 1294 blanks (less than 1%). The results were verified and not considered significant.

In 2018, third-party check assays are on pulps from the primary laboratory that are re-assayed by a third-party laboratory, that is, AGAT pulps were re-assayed by ALS and vice versa. In 2019, ALS was the primary laboratory and SGS was used for the third-party check assays.

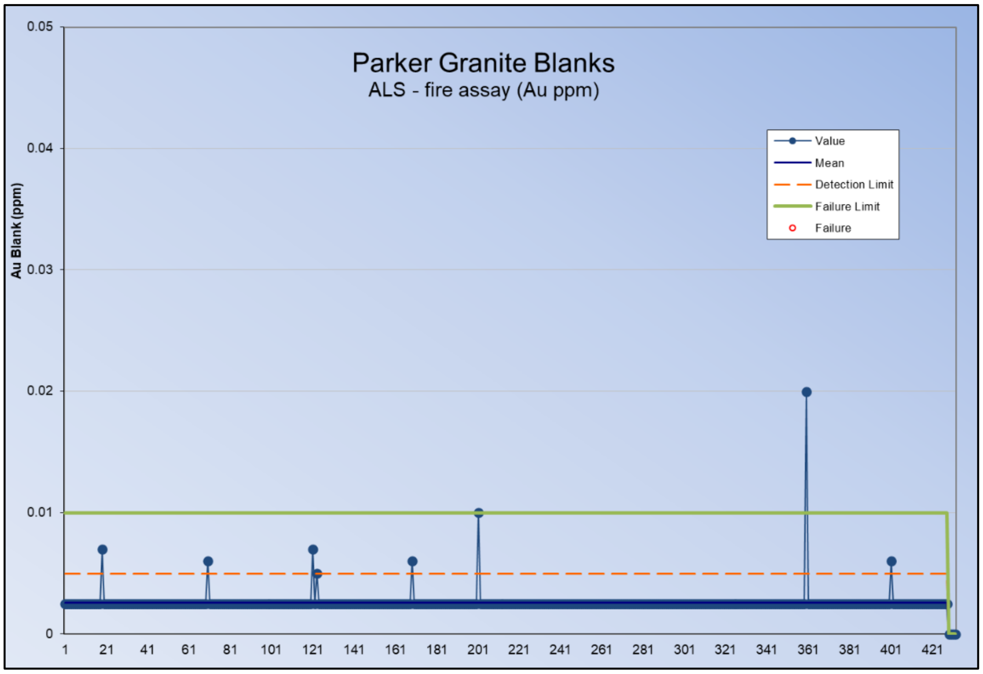
Table 11-5 shows the results of the blanks used in the 2018 – 2019 drilling. Figure 11-1 and Figure 11-2 present plots for fire assay blanks and metallic sieve assay blanks, respectively.

Table 11-5: Blanks Values, 2018 – 2019 Drilling

Troilus Number	Total	Failures	Comment
B1	158	1	
B2	122	2	
B3	194	2	
B4	21	1	
B5	255	1	
B6	97	1	
B7	40	1	
BP	428	1	ALS fire assay
BP	730	1	ALS metallic sieve
BSS	25	0	-

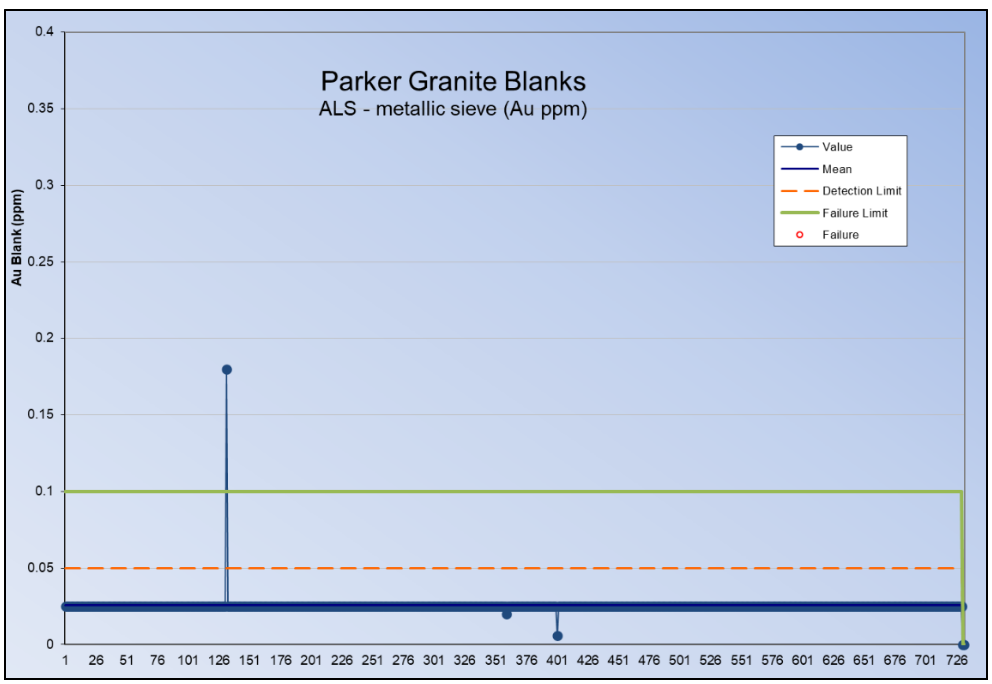
NOTE: BP -Parker Granite Coarse Blank

Figure 11-1: BP Blanks (fire assay) – Gold (ppm Au); 2018-2019 Drilling



Source: AGP (2020b)

Figure 11-2: BP Blanks (fire assay) – Gold (ppm Au); 2018-2019 Drilling



Source: AGP (2020b)

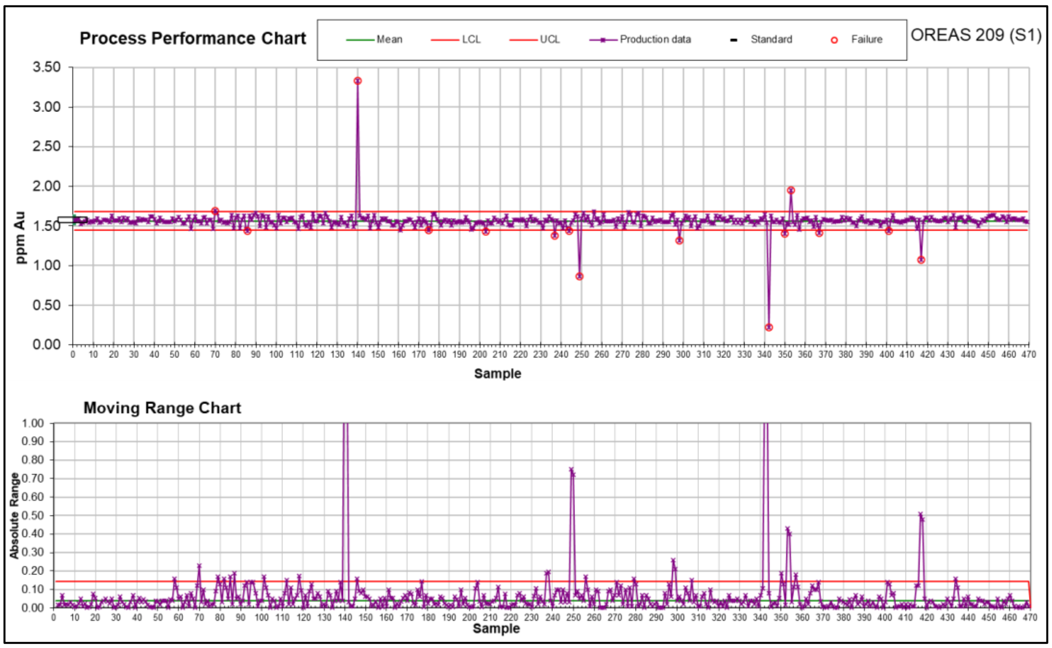
Certified Standard Materials

Table 11-6 presents the results of the CRMs used in the 2018-2019 drilling. Figure 11-3 presents accuracy plot for gold from the 2018 and 2019 drilling.

Table 11-6: CRM Results, 2018 – 2019 Drilling

CRM	Recommended Value	Standard Deviation	Number of Samples	Number of Failures	Percent Failure
OREAS 209 (S1) ppm Au	1.580	0.044	469	15	3.2%
OREAS 215 (S2) ppm Au	3.540	0.097	329	5	1.5%
OREAS 217 (S3) ppm Au	0.338	0.010	500	23	4.6%
OREAS 92 (S4) %Cu	0.229	0.010	32	1	3.1%
OREAS 922 (S5) %Cu	0.212	0.044	479	38	7.9%
OREAS 922 (S5) ppm Ag	0.888	0.109	328	15	4.6%

Figure 11-3: Standard OREAS 209 – Gold Accuracy Plot



Source: AGP (2020b)

11.4.3 QA/QC (2019-2020)

During the 2019 – 2020 drill program on the SW Zone, Troilus continued with the same QA/QC protocols in place: including blank sample materials and CRMs. Table 11-7 shows a summary of the QA/QC samples submitted during the drilling program.

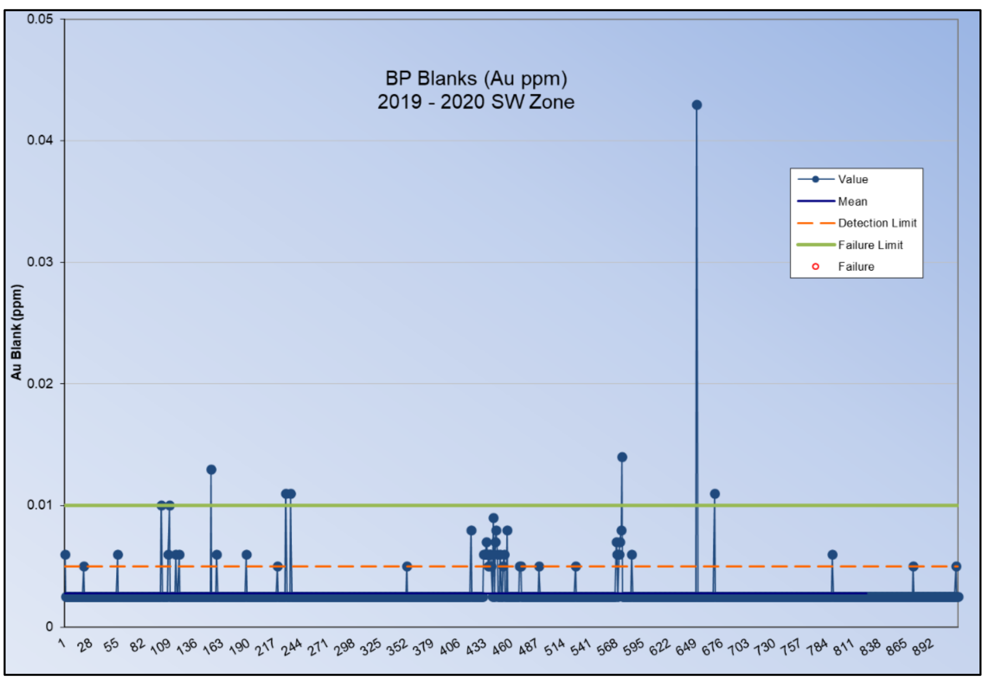
Table 11-7: Summary of Troilus QA/QC Program, 2019 – 2020

Description	Number of Samples (% of database)
Total Number of Samples	21,268
Number of Control Samples	743 (8.7%)
Distribution	
Blanks (BP)	918 (4%)
Blanks (BP)	918
Lab Duplicates	1,701 (8%)
CRM samples	972 (5%)
OREAS 209 (S1)	227
OREAS 215 (S2)	207
OREAS 217 (S3)	240
OREAS 922 (S5)	223
OREAS 239 (S6)	45
OREAS 235 (S7)	26
OREAS 153b (S9)	2

Blanks

During the 2019 – 2020 drilling on the SW Zone, only 6 failures occurred out of 918 blank samples. The results were five samples with less than 0.015 ppm Au, and one sample at 0.043 ppm Au. These were determined not to have a significant impact on the sample batches and were ignored. Figure 11-4 presents the plots for the gold assay blanks.

Figure 11-4: BP Blanks – Gold (ppm Au); 2019 – 2020 Drilling



Source: AGP (2022)

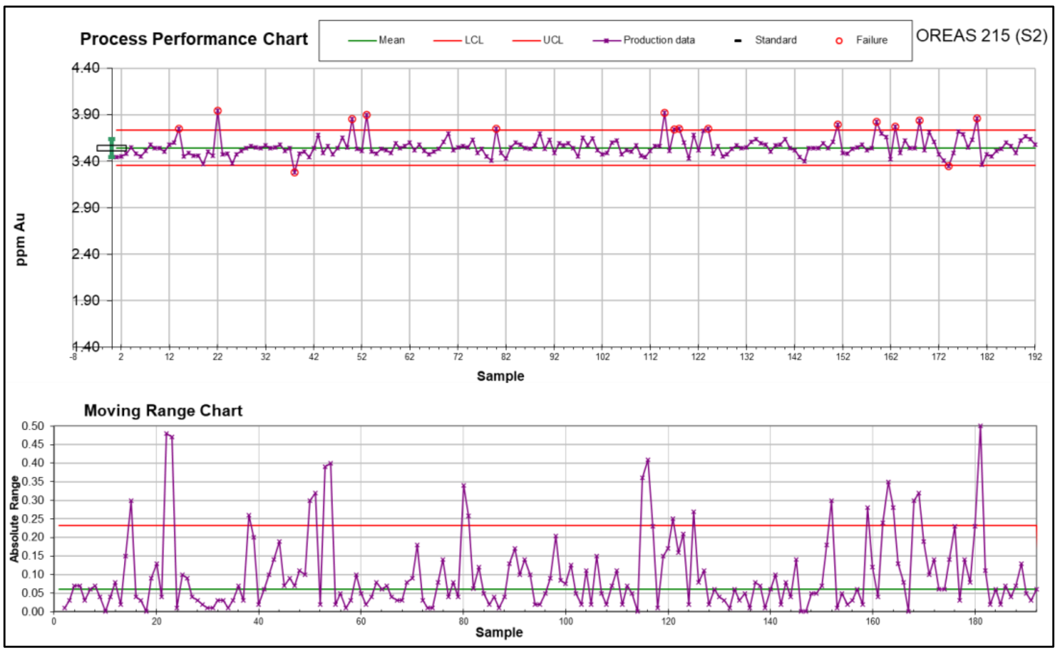
Certified Standard Materials

Table 11-8 presents the results of the CRMs used in the 2019-2020 drilling on the SW Zone. Figure 11-5 presents the accuracy plot for gold for CRM S2 (OREAS 215).

Table 11-8: CRM Results, 2019 – 2020 Drilling – SW Zone

CRM	Recommended Value	Standard Deviation	Number of Samples	Number of Failures	Percent Failure
OREAS 209 (S1) ppm Au	1.58	0.044	217	7	3.2%
OREAS 215 (S2) ppm Au	3.54	0.097	192	16	7.7%
OREAS 217 (S3) ppm Au	0.338	0.010	230	23	10.0%
OREAS 922 (S5) %Cu	0.212	0.009	210	1	0.5%
OREAS 922 (S5) ppm Ag	0.888	0.109	210	12	5.7%
OREAS 239 (S6) ppm Au	3.55	0.086	29	0	-
OREAS 935 (S7) ppm Au	1.59	0.038	14	0	-

Figure 11-5: Standard OREAS 215 (S2)– Gold Accuracy Plot



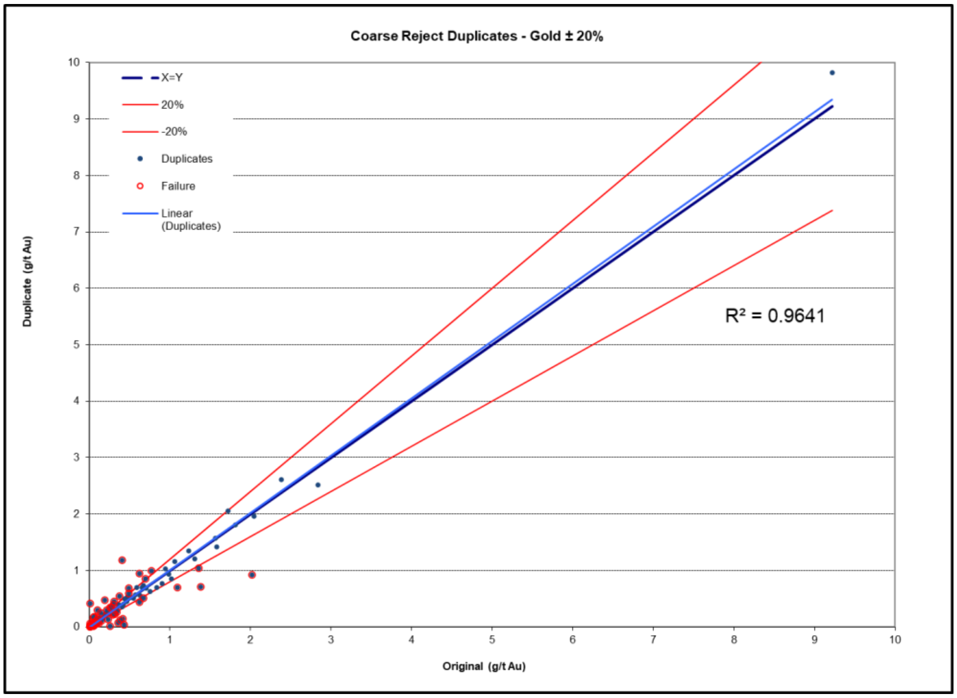
Source: AGP (2022)

Duplicates

During the 2020 drill program on the SW Zone, duplicate samples were conducted on the pulps and rejects returned to Troilus post analysis. The samples were nominally selected based on mineralized domains and the pulps and rejects were sent to either ALS or SGS for analysis.

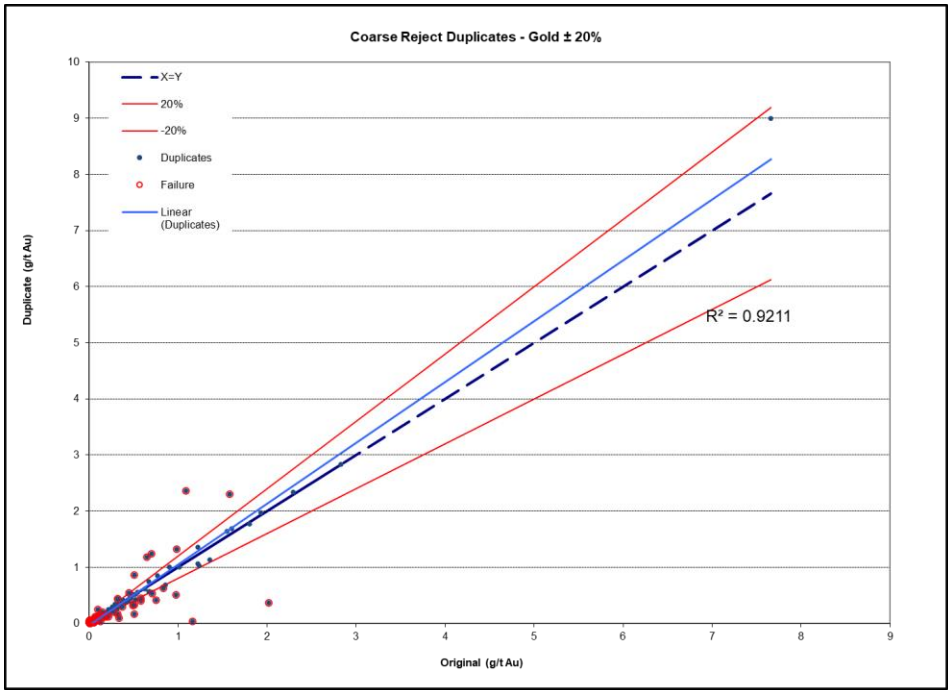
Figure 11-6 and Figure 11-7 show the duplicate control plots for samples sent to ALS and SGS, respectively.

Figure 11-6: Duplicates Control Chart (ALS Rejects) – Gold Values; 2020 SW Zone



Source: AGP (2022)

Figure 11-7: Duplicates Control Chart (SGS Rejects) – Gold Values; 2020 SW Zone



Source: AGP (2022)

11.4.4 QA/QC (2021-2022)

The analytical quality control data produced during the 2021 and 2022 drilling programs was reviewed by AGP. It should be noted that the 2021 QA/QC data includes: one drill hole from Z87 completed in 2020; and four drill holes from J Zone completed in 2020.

The QA/QC program implemented by Troilus is a continuation of the previous program initiated since 2018 for each drilling campaign for the Z87 Zone, J Zone, and SW Zone. The QA/QC samples included: coarse blanks, CRMs, coarse reject, and pulp duplicates.

Table 11-9 and Table 11-10 show a summary of the QA/QC samples submitted during the 2021 and 2022 drilling programs carried out on the Project.

Table 11-9: Summary of Troilus QA/QC Program, 2021

Description	Z87 Number of Samples (% of database)	J Number of Samples (% of database)	SW Number of Samples (% of database)
Total Number of Samples	6,219	9,899	48,132
Number of Control Samples	1,197 (19%)	3,748 (40%)	6,551 (13%)
Distribution			
Blanks	307 (5%)	1,119 (11%)	2,094 (4%)
Blanks (BP)	251	716	466
Blanks (BSS)	41	403	1,628
Blanks (B0, B1, B2, B3, B4, B5)	15	-	-
Lab Duplicates	564 (9%)	1,715 (17%)	2,088 (4%)
CRM samples	326 (5%)	1,173 (12%)	2,149 (5%)
OREAS 209 (S1)		10	4
OREAS 215 (S2)	2	10	5
OREAS 217 (S3)	49	103	20
OREAS 922 (S5)	71	180	99
OREAS 239 (S6)	59	178	98
OREAS 235 (S7)	84	204	227
OREAS 231 (S8)	44	137	178

Note: Z87 Zone includes one drill hole from 2020.
J Zone includes four drill holes from 2020.

Table 11-10 Summary of Troilus QA/QC Program, 2022

Description	Z87 Number of Samples (% of database)	SW Number of Samples (% of database)
Total Number of Samples	4,246	54,556
Number of Control Samples	584 (14%)	2,889 (5%)
Distribution		
Blanks	294 (7%)	1,445 (3%)
Blanks (BSS)	294	1,444
Blanks (B0)	-	1
Lab Duplicates	-	-
CRM samples	290 (7%)	1,444 (2%)
OREAS 209 (S1)	1	-
OREAS 153b (S9)	-	626
OREAS 254b (S10)	149	729
OREAS 506 (S14)	140	89

Blanks

A total of 5,262 coarse blanks were inserted by Troilus personnel to monitor grade contamination during the 2021 and 2022 drill programs, up to May 2022. The two main blank materials used were the Parker Lake Granite (BP) and the silica sand (BSS). The quality control performance of these blank samples was reviewed by AGP. Table 11-11 shows the results of the blank samples by zone.

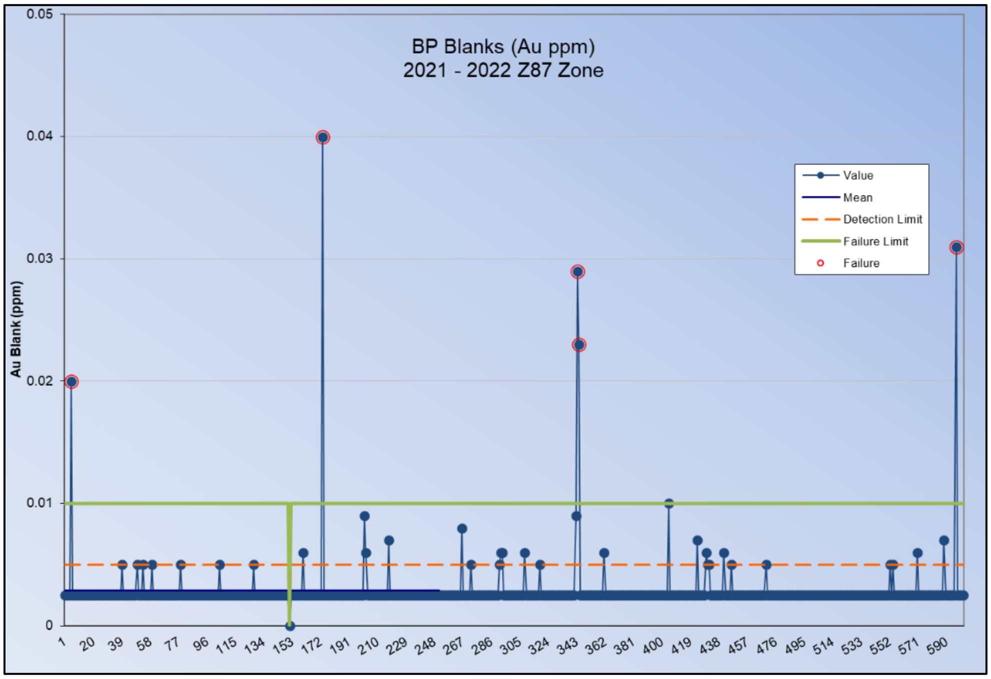
Blank materials were considered failures when the returned gold value exceeded 10 times the lower detection limit, that is, less than 0.005 ppm Au) of the analytical method. Blank material results are considered excellent since the control plots show no contamination on blank materials submitted within the mineralized samples batches

Figure 11-8 to Figure 11-10 show the results of the BP blank materials for gold for the Z87, J and SW Zone, respectively.

Table 11-11: Summary of Blanks Performance, 2021-2022 – All Zones

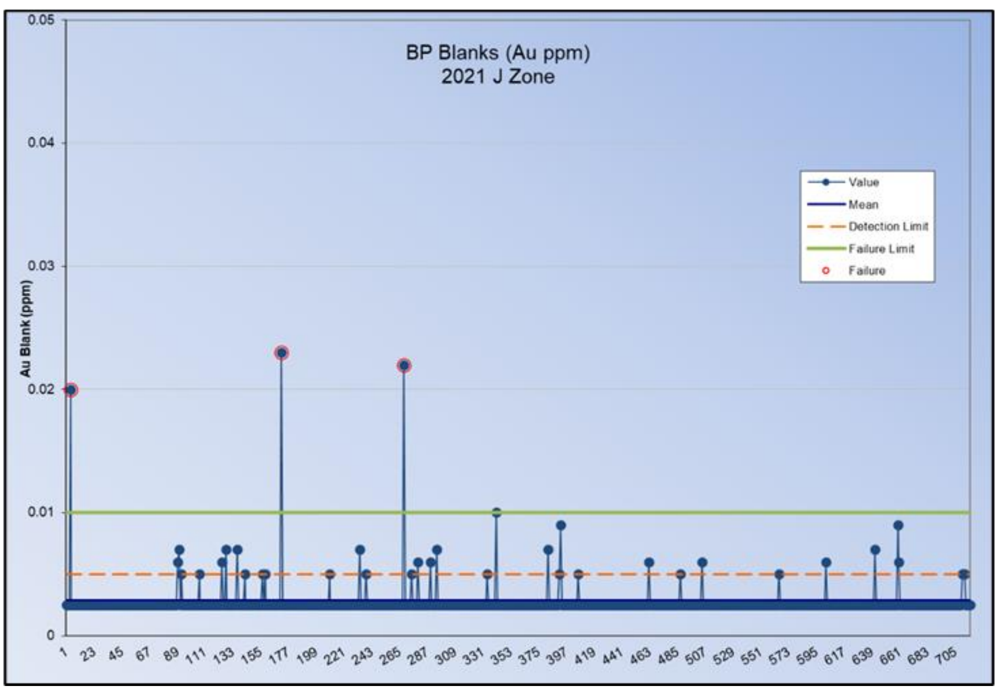
Blank Material	Code	No. of Samples	No. of Failures	Percent Failure
Z87 Zone				
Parker Lake Granite outcrop	BP	251	4	0.7 %
Silica sand (Lac à la Croix)	BSS	338	3	0.9 %
Other Blanks	B0, B1, B2, B3, B4, B5	15	0	-
J Zone				
Parker Lake Granite outcrop	BP	716	2	0.3 %
Silica sand (Lac à la Croix)	BSS	403	1	0.2 %
SW Zone				
Parker Lake Granite outcrop	BP	466	2	0.4 %
Silica sand (Lac à la Croix)	BSS	3072	20	0.7 %
Other Blanks	B0	1	0	-

Figure 11-8: BP Blanks – Gold (ppm Au); 2021 – 2022 Drilling – Z87 Zone



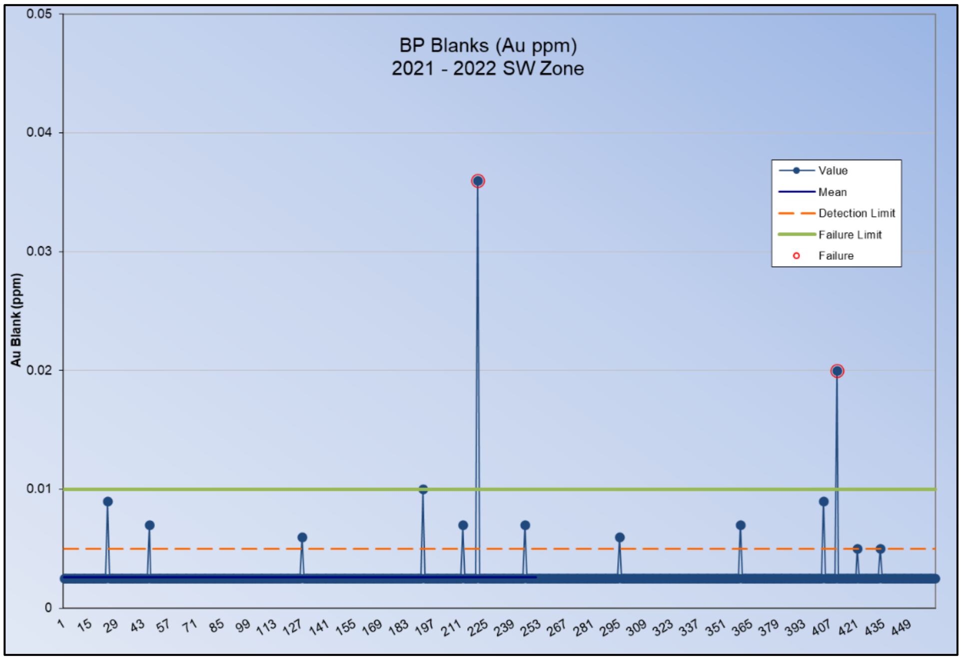
Source: AGP (2022)

Figure 11-9 BP Blanks – Gold (ppm Au); 2021 – 2022 Drilling – Z87 Zone



Source: AGP (2022)

Figure 11-10 BP Blanks – Gold (ppm Au); 2022 Drilling – SW Zone



Source: AGP (2022)

Certified Standard Materials

Table 11-12 to Table 11-14 represent the accuracy performance results of CRM as implemented by Troilus during 2021-2022 drilling programs performed on J, Z87, and SW Zones, respectively. CRMs were considered as failures when a gold or copper result exceeded three times the standard deviation (±3 SD) beyond the certified expected value. Most of CRM failure results are associated with the insertion of the wrong standard sample number.

Table 11-12 CRM Results of the 2021-2022 Drilling – Z87 Zone

CRM	Recommended Value	Standard Deviation	Number of Samples	Number of Failures	Percent Failure
OREAS 215 (S2) ppm Au	3.54	0.097	2	0	-
OREAS 217 (S3) ppm Au	0.338	0.010	49	2	4.1 %
OREAS 922 (S5) %Cu	0.212	0.009	71	23	32.4 %
OREAS 239 (S6) ppm Au	3.55	0.086	58	1	1.7 %
OREAS 235 (S7) ppm Au	1.59	0.038	84	1	1.2 %
OREAS 231 (S8) ppm Au	0.542	0.015	44	2	4.7%
OREAS 153b (S9) ppm Au	0.313	0.009	5	0	-
OREAS 254b (S10) ppm Au	2.53	0.061	150	9	6.0 %

Table 11-13 CRM Results of the 2021-2022 Drilling – J Zone

CRM	Recommended Value	Standard Deviation	Number of Samples	Number of Failures	Percent Failure
OREAS 209 (S1) ppm Au	1.58	0.044	10	0	-
OREAS 215 (S2) ppm Au	3.54	0.097	10	1	10.0 %
OREAS 217 (S3) ppm Au	0.338	0.010	97	5	5.2 %
OREAS 922 (S5) %Cu	0.212	0.009	181	12	6.7 %
OREAS 239 (S6) ppm Au	3.55	0.086	178	9	5.1 %
OREAS 235 (S7) ppm Au	1.59	0.038	204	5	2.5 %
OREAS 231 (S8) ppm Au	0.542	0.015	137	2	1.5 %
OREAS 153b (S9) ppm Au	0.313	0.009	151	10	6.6 %
OREAS 254b (S10) ppm Au	2.53	0.061	137	6	4.4 %

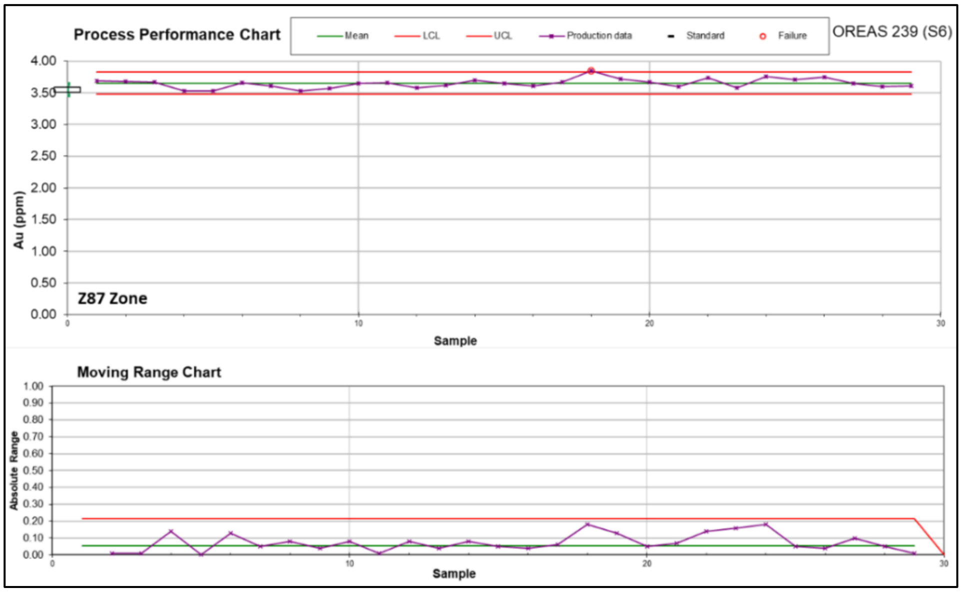
Table 11-14CRM Results of the 2021-2022 Drilling – SW Zone

CRM	Recommended Value	Standard Deviation	Number of Samples	Number of Failures	Percent Failure
OREAS 209 (S1) ppm Au	1.58	0.044	4	1	25.0 %
OREAS 215 (S2) ppm Au	3.54	0.097	5	0	-
OREAS 217 (S3) ppm Au	0.338	0.01	20	0	-
OREAS 922 (S5) %Cu	0.212	0.009	99	4	4.0 %
OREAS 239 (S6) ppm Au	3.55	0.086	97	5	5.2 %
OREAS 235 (S7) ppm Au	1.59	0.038	227	3	1.3 %
OREAS 231 (S8) ppm Au	0.542	0.015	178	6	3.4 %
OREAS 153b (S9) ppm Au	0.313	0.009	1,402	9	0.7 %
OREAS 254b (S10) ppm Au	2.53	0.061	1,419	39	2.7%
OREAS 506 (S14) ppm Au	0.364	0.010	84	2	2.4 %

Figure 11-11 to Figure 11-13 show the control plots for gold results for CRM OREAS 239 inserted within Z87, J and SW Zones, respectively, for sample batches collected in 2021 to 2022. Overall, the results of this CRM show a good performance. Several 'failures' appear to be mislabelled blanks.

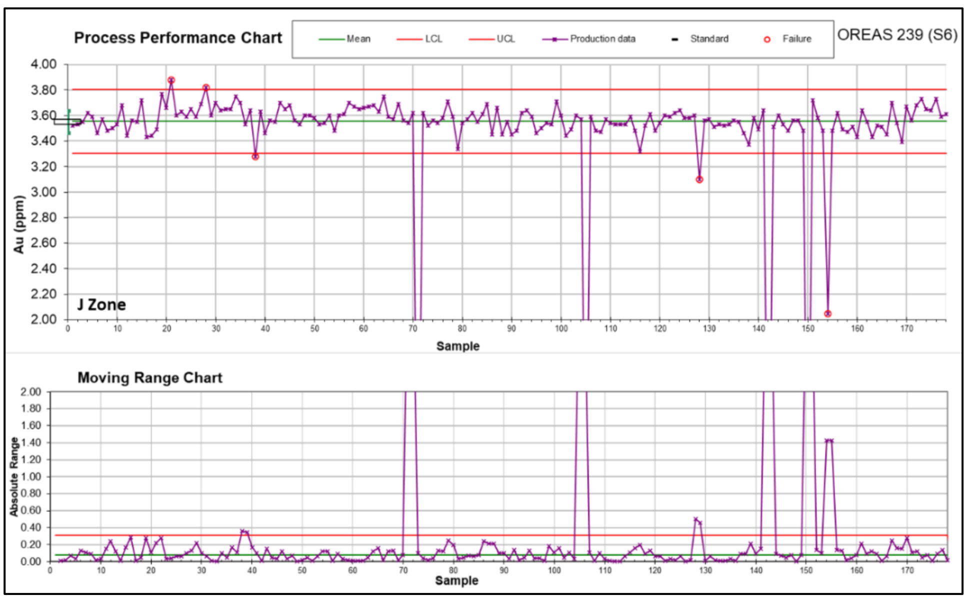
Figure 11-14 to Figure 11-16 show the control plots for the copper results for CRM OREAS 922., inserted within Z87, J and SW Zones, respectively, for sample batches collected in 2021 to 2022. Several 'failures' appear to be mislabelled blanks.

Figure 11-11 Standard OREAS 239 (S6) Control Chart – Gold Control Plot; 2021-2022; Z87 Zone



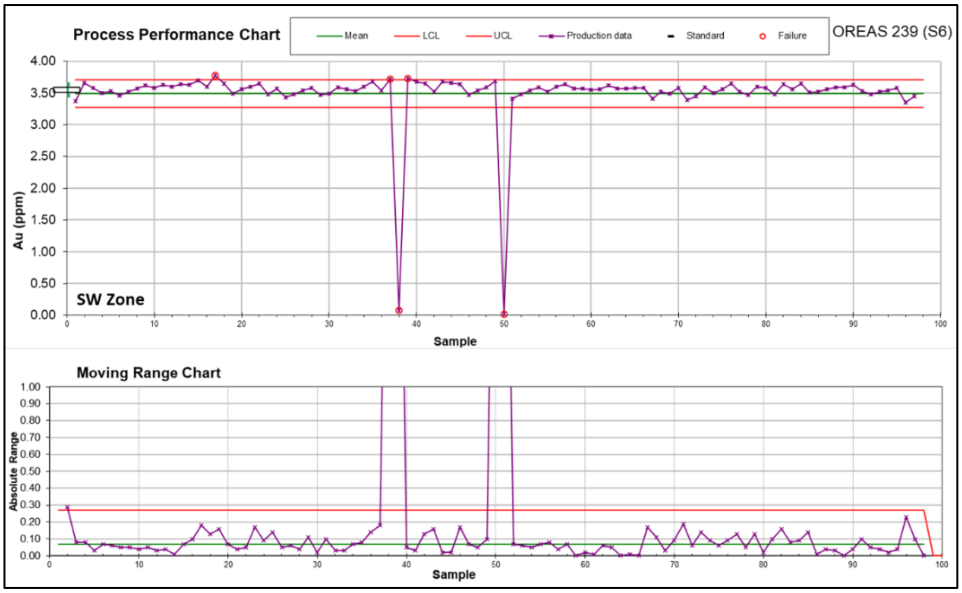
Source: AGP (2022)

Figure 11-12 Standard OREAS 239 (S6) Control Chart – Gold Control Plot; 2021-2022; J Zone



Source: AGP (2022)

Figure 11-13 Standard OREAS 239 (S6) Control Chart – Gold Control Plot; 2021-2022; SW Zone

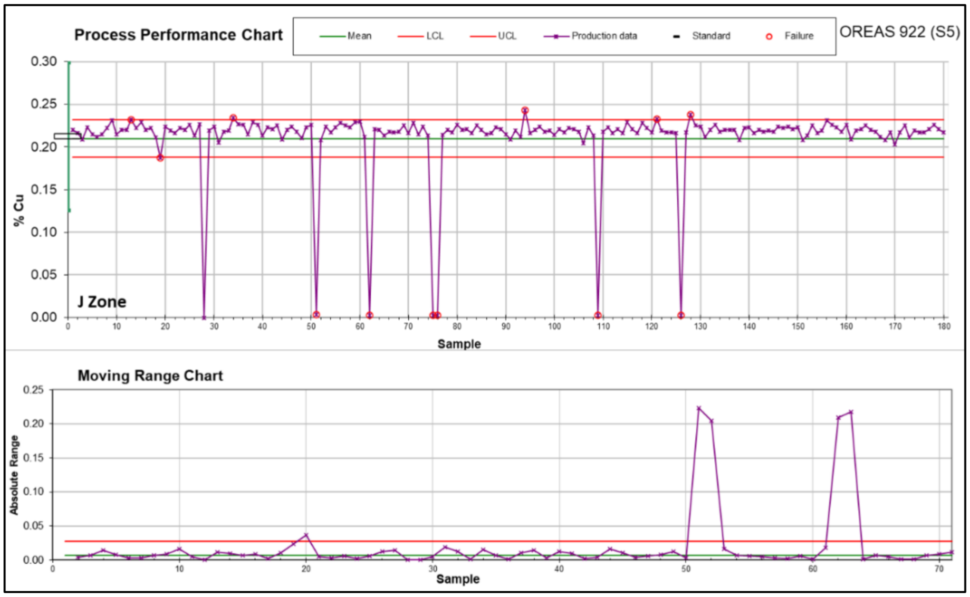


Source: AGP (2022)

Figure 11-14 Standard OREAS 922 Control Chart – Copper Control Plot; 2021-2022; Z87 Zone

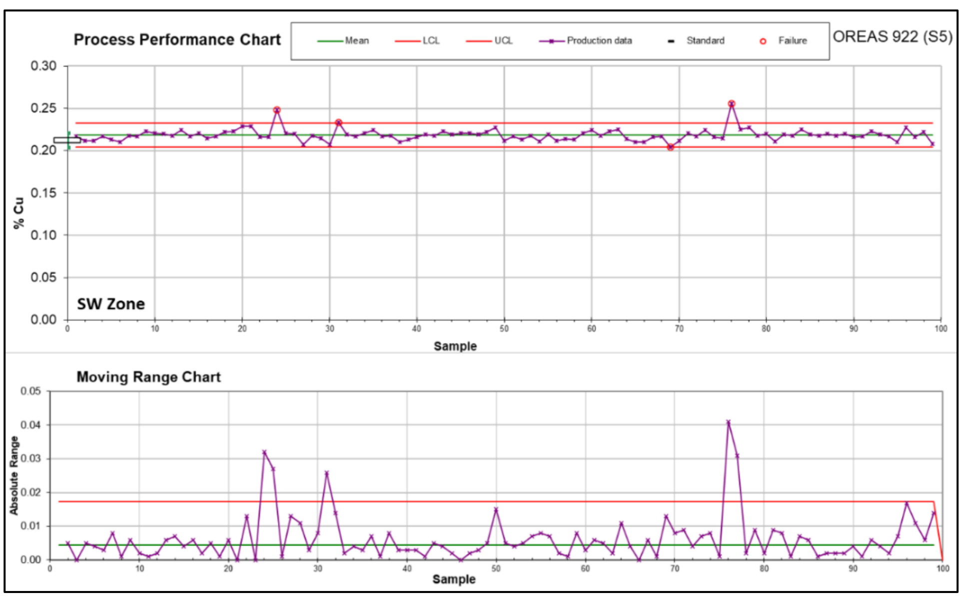
Source: AGP (2022)

Figure 11-15 Standard OREAS 922 Control Chart – Copper Control Plot; 2021-2022; J Zone



Source: AGP (2022)

Figure 11-16 Standard OREAS 922 Control Chart – Copper Control Plot; 2021-2022; SW Zone



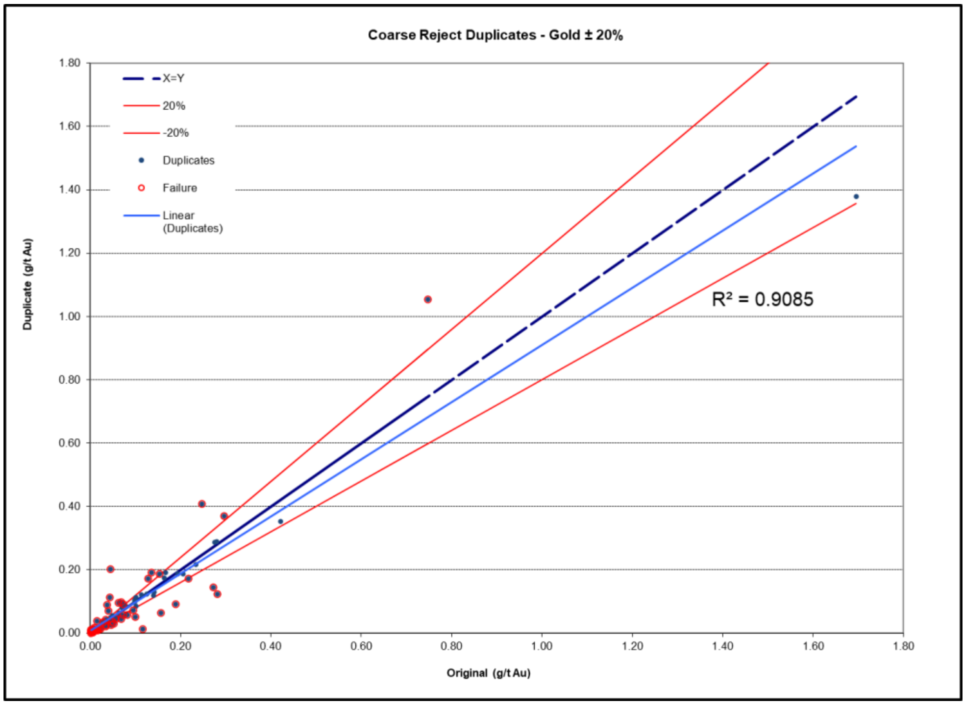
Source: AGP (2022)

Duplicates

During the 2021 and 2022 drill programs, duplicate samples were conducted on the pulps and rejects returned to Troilus post analysis. The samples were nominally selected based on mineralized domains and the pulps and rejects were sent to either ALS or SGS for analysis.

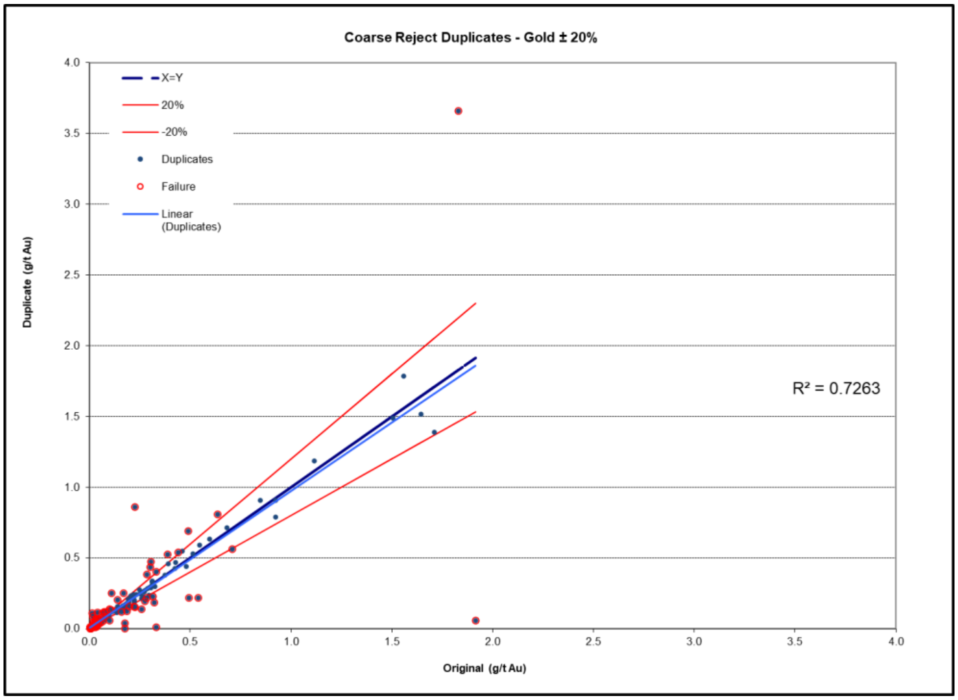
Figure 11-17 to Figure 11-19 show the duplicate control plots of Rejects sent to ALS for Z87, J and SW Zones, respectively.

Figure 11-17 Duplicates Control Chart (ALS Rejects) – Gold Values; 2021 – Z87 Zone



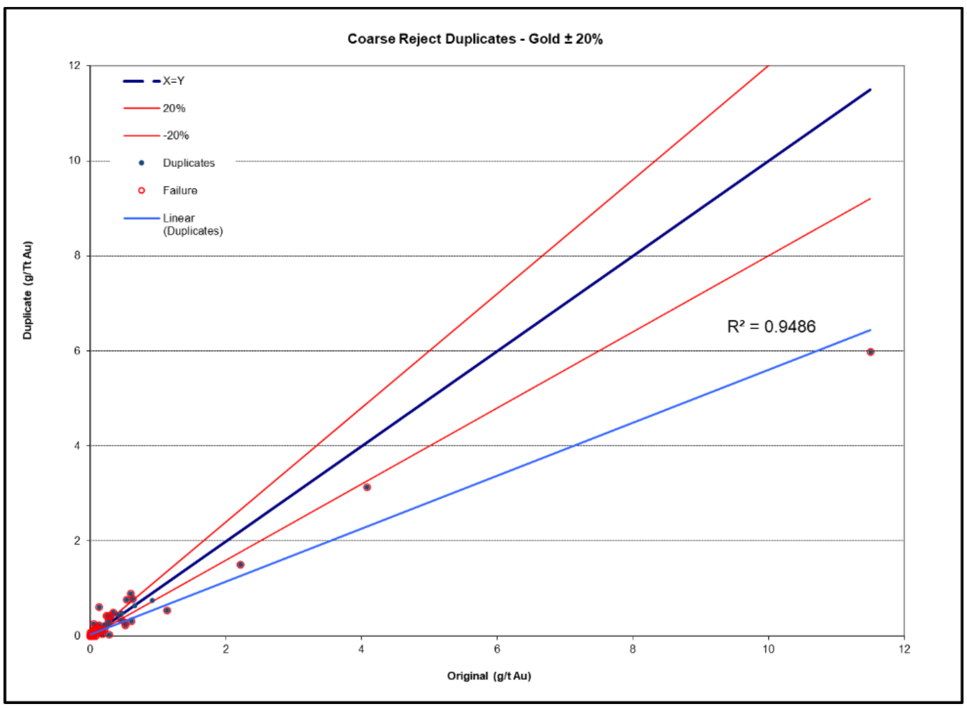
Source: AGP (2022)

Figure 11-18 Duplicates Control Chart (ALS Rejects) – Gold Values; 2021 – J Zone



Source: AGP (2022)

Figure 11-19 Duplicates Control Chart (ALS Rejects) – Gold Values; 2021 – SW Zone



Source: AGP (2022)

11.4.5 QA/QC (2022-2023)

The analytical quality control data produced during the 2022 and 2023 drilling programs, from June 2022 to 31 August 2023, was reviewed by AGP. The QA/QC program implemented by Troilus is a continuation of the previous program initiated since 2018 for each drilling campaign and includes drilling completed on the recently developed X22 Zone. The QA/QC samples included: coarse blanks, CRMs, coarse reject, and pulp duplicates.

Table 11-15 show a summary of the QA/QC samples submitted during the 2022 and 2023 drilling programs carried out on the Project.

Table 11-15: Summary of Troilus QA/QC Program, 2022-2023

Description	All Zones Number of Samples (% of database)
Total Number of Samples	88,707
Number of Control Samples	18,863 (21%)
Distribution	
Blanks	12,309 (14%)
Blanks (B1)	158
Blanks (B2)	122
Blanks (B3)	194
Blanks (B4)	21
Blanks (B5)	269
Blanks (B6)	97
Blanks (B7)	40
Blanks (BP)	3,608
Blanks (BSS)	7,800
Duplicates	798 (2022) + 1,687 (2023); (5%)
CRM samples	4,069 (5%)
OREAS 153b (S9)	128
OREAS 254b (S10)	2,059
OREAS 506 (S14)	1,882

Blanks

A total of 12,309 coarse blanks were inserted by Troilus personnel to monitor grade contamination during the 2022 and 2023 drill programs, up to August 2023. The two main blank materials used were the Parker Lake Granite (BP) and the silica sand (BSS). The quality control performance of these blank samples was reviewed by AGP. Table 11-16 shows the results of each blank sample.

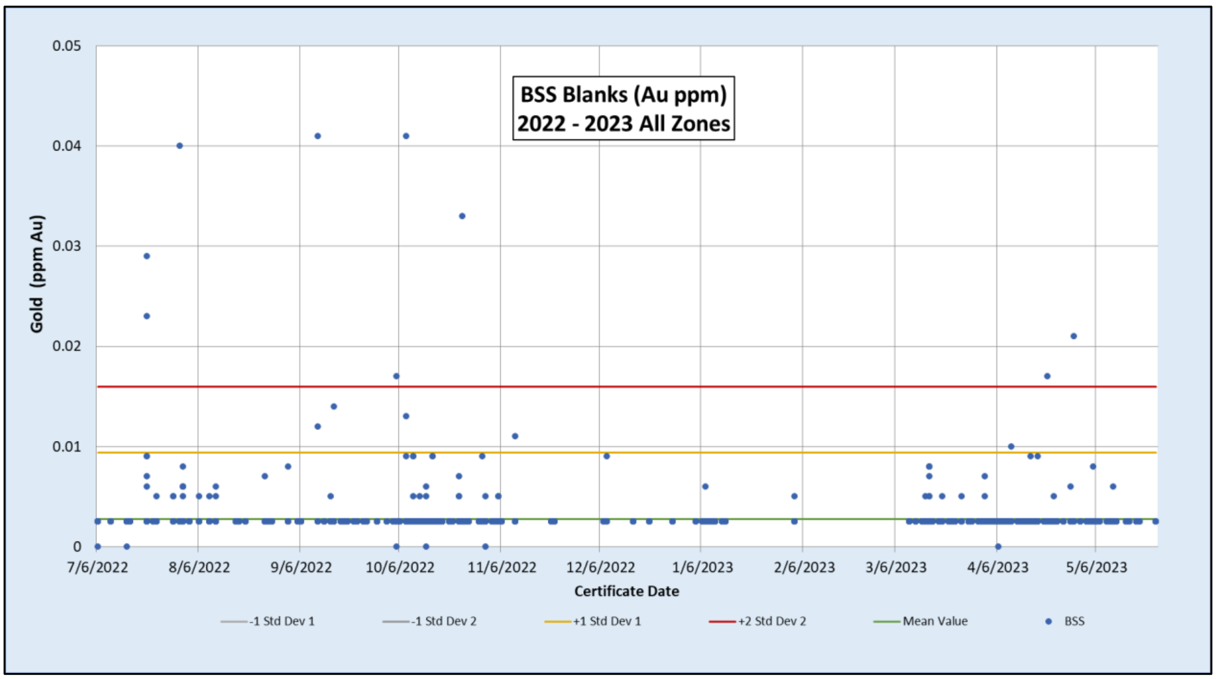
Blank materials were considered failures when the returned gold value exceeded twice the standard deviation, or greater than 0.016 ppm Au. Blank material results are considered excellent since the control plots show few failures of blank materials submitted within the mineralized samples batches.

Figure 11-20 show the results of the BSS blank material for gold.

Table 11-16: Summary of Blanks Performance, 2022-2023 – All Zones

Blank Material	Code	Number of Samples	Number of Failures	Percent Failure
Blanks (B1)	(B1)	158	1	0.6%
Blanks (B2)	(B2)	122	5	4.1%
Blanks (B3)	(B3)	194	7	3.6%
Blanks (B4)	(B4)	21	2	9.5%
Blanks (B5)	(B5)	269	9	3.3%
Blanks (B6)	(B6)	97	3	3.1%
Blanks (B7)	(B7)	40	0	0.0%
Parker Lake Granite outcrop	(BP)	3608	11	0.3%
Silica sand (Lac à la Croix)	(BSS)	7800	27	0.3%

Figure 11-20: BSS Blanks – Gold (ppm Au); 2022 – 2023 Drilling – All Zones



Source: Troilus (2023)

Certified Standard Materials

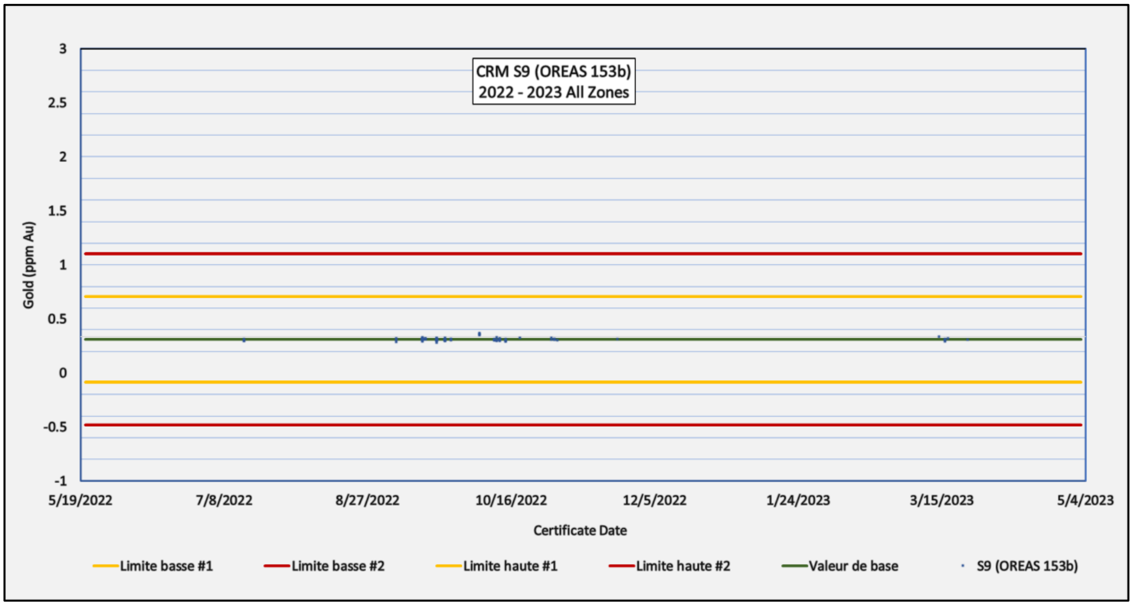
Table 11-17 presents the accuracy performance results of CRM as implemented by Troilus during 2022-2023 drilling programs. CRMs were considered as failures when a gold or copper result exceeded three times the standard deviation (±3 SD) beyond the certified expected value.

Figure 11-21 to Figure 11-23 presents the control plots for the three CRMs used in the 2022 – 2023 drill programs: OREAS 153b (S9), 254b (S14), 506 (S14), respectively.

Table 11-17: CRM Results of the 2022-2023 Drilling

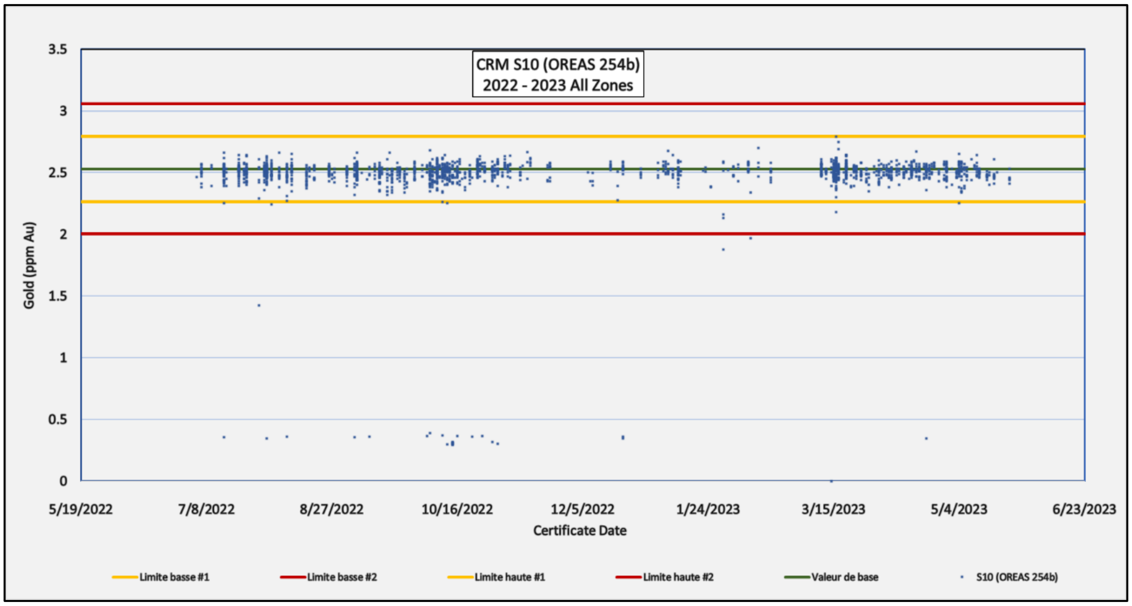
CRM	Recommended Value	Standard Deviation	Number of Samples	Number of Failures	Percent Failure
OREAS 153b (S9) ppm Au	0.313	0.009	128	3	2.3%
OREAS 254b (S10) ppm Au	2.53	0.061	2059	5	0.2%
OREAS 506 (S14) ppm Au	0.364	0.013	1882	17	0.9%

Figure 11-21: Standard OREAS 153b (S9) Control Chart – Gold Control Plot; 2022-2023– All Zones



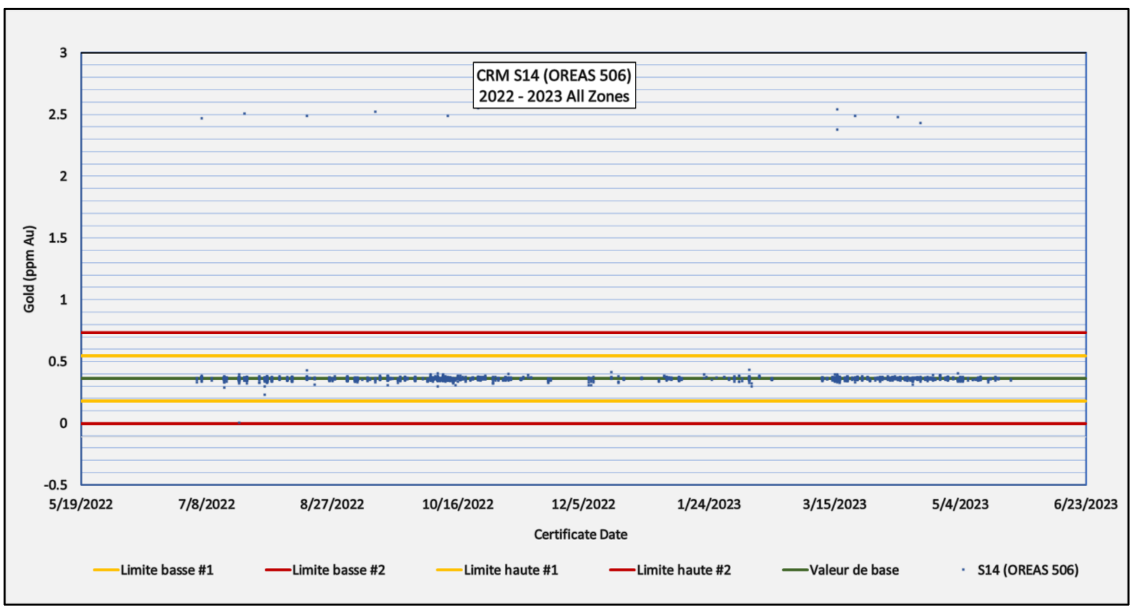
Source: Troilus (2023)

Figure 11-22: Standard OREAS 254b (S10) Control Chart – Gold Control Plot; 2022-2023– All Zones



Source: Troilus (2023)

Figure 11-23: Standard OREAS 506 (S14) Control Chart – Gold Control Plot; 2021-2022– All Zones



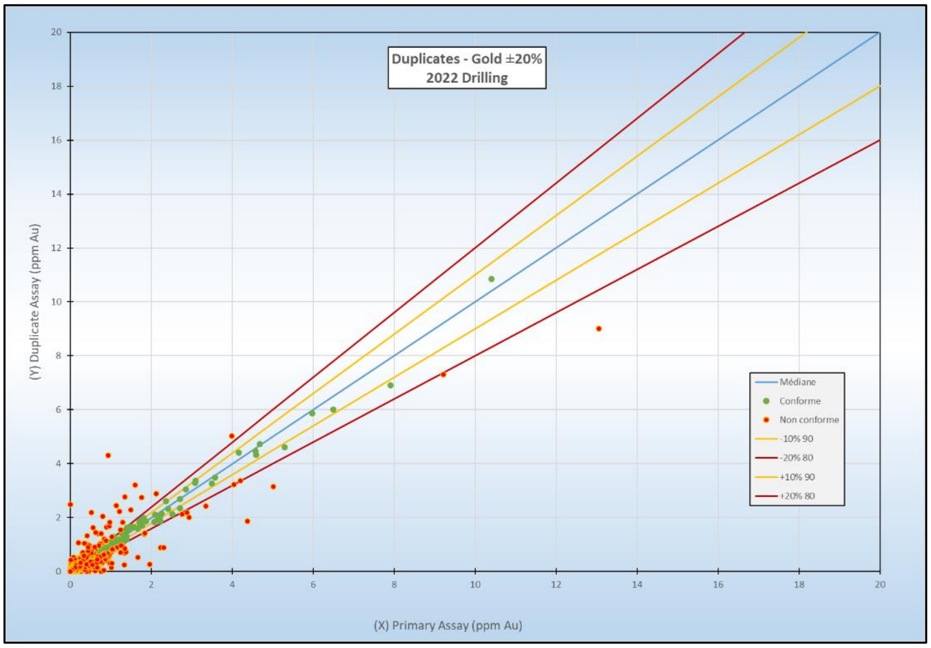
Source: Troilus (2023)

Duplicates

During the 2022 – 2023 drill programs, duplicate samples were conducted on the pulps and rejects returned to Troilus post analysis. The samples were nominally selected based on mineralized domains and the pulps and rejects were sent to either ALS or SGS for analysis.

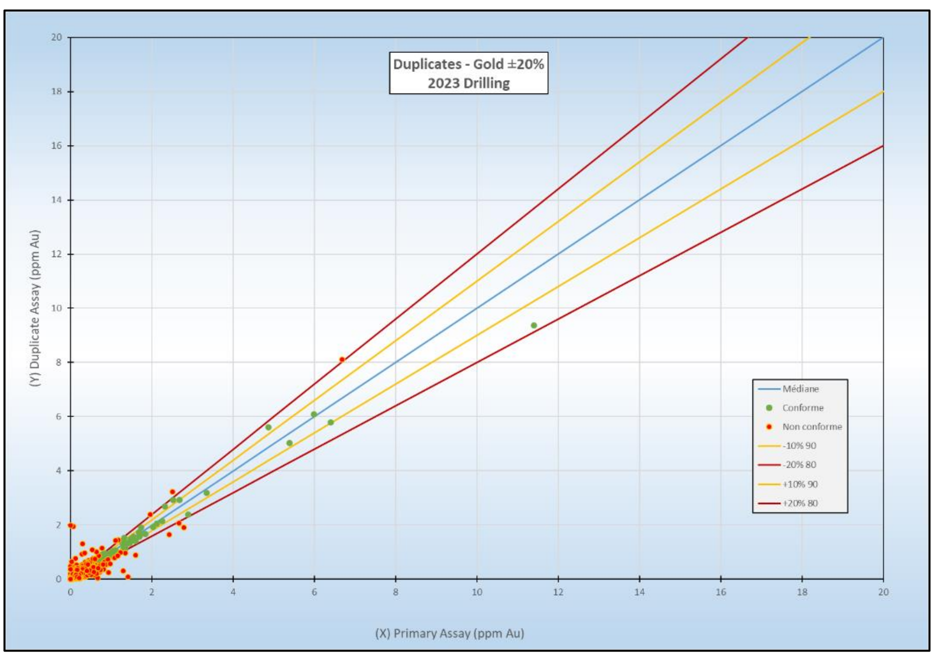
Figure 11-24 and Figure 11-25 show the duplicate control plots for the 2022 and 2023 drilling, respectively.

Figure 11-24: Duplicates Control Chart – Gold Values; 2022 Drilling



Source: Troilus (2023)

Figure 11-25: Duplicates Control Chart – Gold Values; 2023 Drilling



Source: Troilus (2023)

11.5 Databases

Troilus maintains their exploration data in a Geotic database and employs a database manager to maintain the integrity of the database. Only senior level technicians have access to the database.

11.6 Sample Security

Samples are kept secure in the core logging and sampling facility until they are shipped. Troilus maintains a strict chain of custody of their samples from core shed to the transport company to the assay laboratory.

Pulps and rejects are stored on site, near the core logging facilities, in sea containers and custom-built storage sheds between the sea containers. Core boxes are stored next to the sea containers in covered steel core racks.

11.7 Qualified Person Opinion

AGP reviewed the sample preparation, analytical and security procedures, as well as insertion rates and the performance of blanks and CRMs from Troilus 2018 to 2023 drill holes and considers that the observed failure rates are minor and that no significant bias affected the integrity of the assay data. In the opinion of the QP, the analytical results delivered by the accredited laboratory and the quality of the data for the Z87, J, X22 and SW Zones are in accordance with the industry standards and are sufficiently reliable for mineral resource estimation.

AGP reviewed the QA/QC program and is of the opinion it is in accordance with standard industry practice and CIM Exploration Best Practice Guidelines. Troilus personnel have taken all reasonable measures to ensure the sample analysis completed is accurate and precise. AGP considers the assay results and database acceptable for use in the estimation of mineral resources. AGP recommends implementing a QA/QC check at the reception of each certificate to assure the quality of the results.

It is the opinion of the QP that the preparation and analyses are satisfactory for this type of the deposit and that the sample handling and chain of custody meet or exceed industry standards.

Density measurements collected during the Troilus drilling program are acceptable and satisfactory. AGP recommends that density measurements continue to be collected for all future drill programs.

12 DATA VERIFICATION

12.1 Data Verification, All Zones

AGP received the database containing all drill holes for the Z87 Zone, J Zone, X22 Zone, and SW Zone in a Leapfrog Project that included, but not limited to, collar, survey, assay, and lithology tables. An export of the Geotic database was received for data validation and QA/QC review.

AGP verified approximately 7.5% of the data from the 2021 and 2022 drill programs (approximately 13,000 records out of 175,000) and included data across all four zones. AGP verified approximately 10% of the data from the 2023 drill program mainly from the X22 Zone. The gold, copper, silver assay values, and density values, were compared to the laboratory certificates provided to Troilus by ALS. No errors were found.

The drill holes were also checked visually for any misplaced drill hole collars, erroneous down hole surveys and for any missing or overlapping intervals. No errors were found.

12.2 AGP Site Visit

The most recent site inspection was conducted by Paul Daigle, Principal Resource Geologist with AGP, and QP for this report, from 5 October to 7 October 2022 for two days. The QP was accompanied on the site visit by

- Kyle Frank, géo., Troilus Exploration Manager
- Nic Guest, géo., Troilus Senior Project Geologist
- Konstantin de Maack, Project Geologist, stagiaire
- Nicolas Robert-Potvin, Troilus geotechnician

The site visit included an inspection of core logging and sampling facilities, core storage facilities, verifying drill hole collar coordinates, and reviewing drill core logs against selected drill core.

The QP completed a previous site inspection from 18 February to 20 February 2020, while the 2020 drill program was in progress on the SW Zone.

12.2.1 Logging, Sampling, and Storage Facilities

Drill core for the Project is logged, sampled, and stored temporarily in the rear of a permanent warehouse on the mine site where the front serves as a garage. This facility has a second-floor loft that serves as an office for Troilus geology personnel. The Administrative Centre is situated next to the warehouse and serves administration and additional exploration offices for the Project.

Figure 12-1 shows the warehouse used for core logging and sampling. Figure 12-2 shows the interior of the core logging facility.

Figure 12-1: Drill Core Logging and Sampling Facility

Source: AGP (2022)

Figure 12-2: Drill Core Logging and Sampling Facility

Source: AGP (2022)

The interior the core logging and sampling facility is kept clean and well-maintained. All field and sampling and CRM supplies are kept orderly and organized on shelves and in filing cabinets.

Figures 12-3 shows the Administrative Centre used for administration and exploration offices.

Figure 12-3: Core Logging Facility (centre) and Administrative Centre (right)

Source: AGP (2022)

12.2.2 Drill Hole Core Storage Area and Facilities

The core storage area is situated approximately 300 m west of the core logging facility. Core boxes are stored in tin covered steel racks. The core racks are arranged in a grid pattern and in blocks for easy access. Each block is given a block letter and number, and a record of the location core boxes is kept up to date. Boxes are stored without covers.

Figure 12-4 shows the core racks in the core storage area. Figure 12-5 shows the sea containers and rooms used for storing pulp and rejects after laboratory analysis.

Figure 12-4: Core Storage Area; Core Racks

Source: AGP (2022)

Figure 12-5: Core Storage Area; Pulp and Rejects Storage Facility

Source: AGP (2022)

12.2.3 Drill Hole Collar Locations

Several drill hole collar coordinates were verified at the Z87, J and SW Zones. The locations of the drill hole collars were measured in the field using a hand-held Global Positioning System (GPS) device (Garmin GPS map 62s) using NAD 83 datum, the same datum used by Troilus.

Drill hole collars are capped by an aluminium screw cap that is punched with the drill hole number on top. The drill hole is marked by a 2 m metal rod topped by a metal red-painted flag marked/etched with the drill hole number. In some cases, a 2 m wood stake is planted next to the metal rod, painted orange, and marked with the drill hole the drill hole number. Some wooden stakes have the drill hole number, azimuth, dip, and length written in permanent marker still visible, or with an aluminium tag stapled on with the same information. These rods make the drill hole more easily identifiable and visible above the level of snow in winter.

Figure 12-6 shows drill hole collars for J-22-333 and SW-22-577.

Figure 12-6: Drill Hole Collars for J-22-333 and SW-22-577

Source: AGP (2022)

The collar coordinates measured by AGP fell within a 9 m tolerance of those reported by Troilus. It is the QPs opinion the coordinates are acceptable, given the accuracy of the handheld GPS used to review the drill hole collar locations.

Table 12-1 presents the comparison of the AGP and Troilus drill hole coordinates in the Z87 and J Zones. Table 12-2 presents the comparison of the AGP and Troilus drill hole coordinates in the SW Zone.

Table 12-1: Comparison of Drill Hole Collar Coordinates – Z87 and J Zone

Drill Holes	Troilus Easting (m UTM)	Troilus Easting (m UTM)	AGP Easting (m UTM)	AGP Easting (m UTM	Δ Easting (m)	Δ Northing (m)
Z87-22-420	537214.4	5651828	537212	5651831	3	-3
Z87-22-428	537355.2	5651988	537353	5651990	2	-2
TLG-Z8721-265	537380.9	5651972	537378	5651973	3	-2
Z87-22-424	537319	5651874	537317	5651877	2	-3
Z87-22-259	537553.3	5651790	537551	5651791	3	-1
Z87-22-260	537598	5651802	537596	5651803	2	-1
Z87-22-261	537597.2	5651829	537597	5651829	0	1
Z87-22-262	537655.3	5651827	537652	5651827	4	0
Z87-22-410	536846.1	5651143	536849	5651151	-3	-8
Z87-22-413	536870.3	5651023	536868	5651025	3	-2
Z87-22-431	537103.5	5650526	537102	5650518	2	8
Z87-22-433	537056	5650448	537054	5650452	2	-3
Z87-22-402	536874	5650402	536872	5650403	2	-2
Z87-22-401	536916	5650389	536918	5650392	-2	-3
Z87-22-403	536912.3	5650348	536914	5650350	-2	-2
Z87-22-420	537214.4	5651828	537212	5651831	3	-3
Z87-22-428	537355.2	5651988	537353	5651990	2	-2
J-22-333	536813	5652084	536810	5652078	3	6
J-21-315	536777	5652117	536779	5652119	-2	-2
J-22-332	536752	5652168	536743	5652171	9	-3
TLG-ZJ21-287	536757	5652246	536756	5652248	2	-2
TLG-ZJ21-288	536749	5652312	536749	5652317	0	-4
TLG-ZJ21-289	536775	5652351	536774	5652353	0	-2
TLG-ZJ21-226	536848	5652358	536845	5652360	3	-3
TLG-ZJ21-225	536835	5652303	536833	5652306	2	-3
TLG-ZJ19-110	536851	5652249	536849	5652252	2	-3
TLG-ZJ20-224	536797	5652212	536795	5652214	2	-2
TLG-ZJ19-151	537016	5652434	537016	5652437	0	-3

Table 12-2: Comparison of Drill Hole Collar Coordinates – SW Zone

Drill Holes	Troilus Easting (m UTM)	Troilus Easting (m UTM)	AGP Easting (m UTM)	AGP Easting (m UTM	Δ Easting (m)	Δ Northing (m)
TLG-SW20-180	534188	5647780	534186	5647781	2	-1
TLG-SW20-214	535078	5648401	535078	5648400	-1	1
TLG-SW20-219	535284	5648740	535288	5648740	-3	0
TLG-SW21-218	535219	5648632	535220	5648631	-1	0
TLG-SW21-281	535251	5648682	535255	5648684	-4	-2
TLG-SW21-282	535189	5648561	535189	5648561	-1	0
TLG-SW21-283	535157	5648435	535157	5648436	0	-1
TLG-SW21-284	535161	5648506	535161	5648507	0	-1
TLG-SW21-512	534253	5647771	534252	5647770	1	1
SW-21-217	534121	5647736	534124	5647733	-2	2
SW-21-511	534307	5647710	534307	5647712	0	-2
SW-21-513	534130	5647813	534132	5647810	-2	3
SW-21-589	535333	5648827	535330	5648832	3	-5
SW-22-573	534890	5648057	534890	5648059	0	-3
SW-22-576	534688	5647926	534688	5647927	0	-1
SW-22-577	534692	5647824	534693	5647825	0	-1
SW-22-617	534367	5647776	534367	5647776	0	1
SW-22-621	534460	5647780	534457	5647781	3	-1
SW-22-622	534264	5647668	534263	5647671	1	-3
SW-22-623	534195	5647687	534193	5647689	2	-2
SW-22-624	534117	5647646	534119	5647646	-2	0
SW-22-630	535136	5648370	535135	5648375	1	-5

12.2.4 Drill Hole Log and Drill Core Review

A review of the drill core and drill core logs was made on selected drill core intervals in the Z87, J and SW Zones. The lithology descriptions and sample intervals in the drill logs were compared and found to be consistent. All sample tag numbers in the core boxes match with the intervals in the database.

Table 12-3 lists the selected drill core intervals examined during the site visit.

Table 12-3: Comparison of Drill Hole Collar Coordinates – SW Zone

Zone	Drill Hole	From (m)	To (m)	Interval (m)	Core Boxes
Z87	87-21-408	79.50	84.00	4.50	10
Z87	87-22-415	408.00	421.12	13.12	92-94
Z87	87-22-421	429.00	456.15	27.15	98-103
J	TLG-ZJ21-230	28.27	40.52	12.25	7-10
J	TLG-ZJ21-244	82.25	95.12	12.87	19-21
J	J-21-303	413.21	430.56	17.35	99-102
SW	TLG-ZSW20-204	315.00	314.00	322.10	73-74
SW	TLG-ZSW21-266	168.24	194.25	26.01	39-44
SW	SW-21-501	203.78	216.55	12.77	47-49
SW	SW-21-537	63.27	76.17	12.90	13-16
SW	SW-21-573	29.22	42.00	12.78	7-9
SW	SW-22-641	36.00	44.61	8.61	5-6
SW	SW-22-641	87.15	99.75	12.60	17-19

12.2.5 Independent Samples

There were no independent samples collected during the site visit. Independent samples were collected and analyzed during the February 2020 site inspection (AGP, 2020).

12.3 Qualified Person Opinion

The QP is of the opinion the database is representative and adequate to support the resource estimates for the Troilus deposits. The QP is also of the opinion the core descriptions, sampling procedures, and data entries were conducted in accordance with industry standards.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Samples from the J, 87, Southwest (SW), and X22 zones were submitted to various testing facilities for metallurgical testing in support of the current studies. This report focuses on the more recent testwork from 2021 to 2024. A summary of the historical testwork program is presented in Table 13-1 in Section 13.2.

Recent testwork results were reported in the following documents:

- Memorandum on Sample Assessment, P. Desautels, AGP Mining Consultants Inc., October 2023.
- Secondary Crushing HPGR/BM Circuit - 50,000 tpd, Report No. 7356 Rev 0, Orway Mineral Consultants, March 2024.
- Fragmentation Assessment Troilus Gold, AGP Mining Consultants Inc., November 2024.
- Troilus Gold Project Composite 01 and Composite 02 HPGR Single Pass and Locked Cycle Testwork, Project No. 9318C, KCA, February 2024.
- Comminution Testing Results for Troilus Sample (HPGR Product SMC Tests) Project No. 13134, Hazen Research Inc., January 9, 2024.
- High-Pressure Grinding Abrasions Testwork (Atwal index) for Troilus Project, Sample ID. 23-16563, FLSmidth Mining Technologies GmbH, February 2024.
- Supplemental Metallurgical Testing of the Troilus Gold Project – BL08003, Base met Labs, August 2022.
- Troilus Flotation Testwork Report for J, SW & 87 (3,000 kg sample), SAN 536940 MTR 21 -108, Eriez, August 2022.
- Troilus Flotation Testwork Report for J, SW & 87 (800 kg sample), SAN 5316803 MTR 22 -122, Eriez, October 2023.
- Troilus Flotation Testwork Report for J, SW & 87 (150 kg sample), SAN 577995, Eriez, Nov 2023.
- Troilus Flotation Testwork Report for X22-1 and X22-2 (1500 kg sample), SAN 579517, Eriez, May 2024.
- Gravity Recoverable Test Report for 87, & SW, Project Reference: P-21102, FLSmidth, December 2021.
- Gravity Recoverable Test Report for J Zone, Project Reference: P-21076, FLSmidth, September 2021.
- Gravity Recoverable Test Report for J, SW1, SW2 & 87, Project Reference: P-23014, FLSmidth, July 2023.
- Gravity Recoverable Test Report for X22, Project Reference: P-23059, FLSmidth, May 2024.
- Solid Liquid Separation Testing Report, 88054A, Pocock Industrial Inc. June 2020.
- Email Ref: Acid Base Accounting Testwork, Environment and Permitting, November 2023.

Historically, the J4 Zone and J5 Zone denoted the eastern and western mineralized domains of the J-Zone, respectively. The interpretation of J Zone has changed since 2022, the terms J4 and J5 are referenced as the J Zone.

13.2 Summary of Historical Testing

Numerous metallurgical testwork programs were referenced during the mineral resource estimate study. Table 13-1 shows the testwork programs reviewed during that period.

Table 13-1: Summary Table of Testwork Programs

Program	Comminution	Bench Scale Flotation	Lock Cycle Tests	Gravity Gold Recovery	Pilot Gravity-Flotation	Bottle Rolls	Column Leach
Lakefield SGS 1993-1994	✓	✓		✓	✓	✓	
Lakefield 2003		✓	✓	✓			
Corem 2019						✓	
Corem & Hazen - 2020, 2021	✓						
Kappes, Cassiday & Associates - 2020		✓				✓	✓
Corem 2021		✓	✓	✓	✓	✓	

The details of the metallurgical testwork (1993- 2021) can be found in the published Technical Report and Mineral Resource Estimate on the Troilus Gold-Copper Project, dated October 26, 2023. All referenced testwork documents associated with historical testwork are also listed in that report.

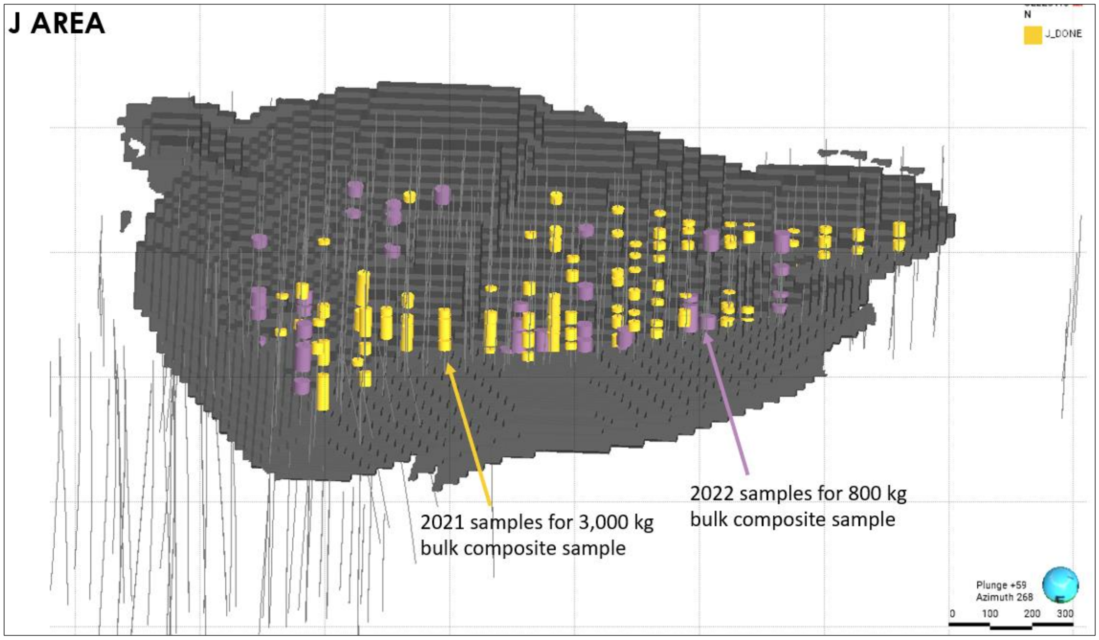
The sections to follow pertain to metallurgical testwork performed on recent samples for J, 87, SW and X22 zones used in the metallurgical evaluation for this report.

13.3 Metallurgical Sample Selection

In 2021, Troilus prepared 3,000 kg representative composite samples from each of the three deposits – J, SW and 87 Zone for metallurgical testwork to support studies leading up to the FS phase. Following the expansion of the resource in 2022, another set of composite samples, 800 kg each, were collected from J, SW and 87 zones to represent the newly discovered resource. In 2023, a new zone called X22 was discovered and new representative 1,500 kg composite samples from this zone were also collected to be part of the metallurgical testwork program. Troilus retained a resource geologist, P. Desautels from AGP, to check sample representativeness for zones J, SW and 87 on the 3,000 and 800 kg samples. The findings from AGP indicated that the sample selection correctly targeted the areas that would form much of the reserve that will ultimately be mined, and the samples are representative of the tonnage distribution. Representativity of X22 samples was not checked as this zone only contributes to 10% of the reserve and will not be mined until after year 15.

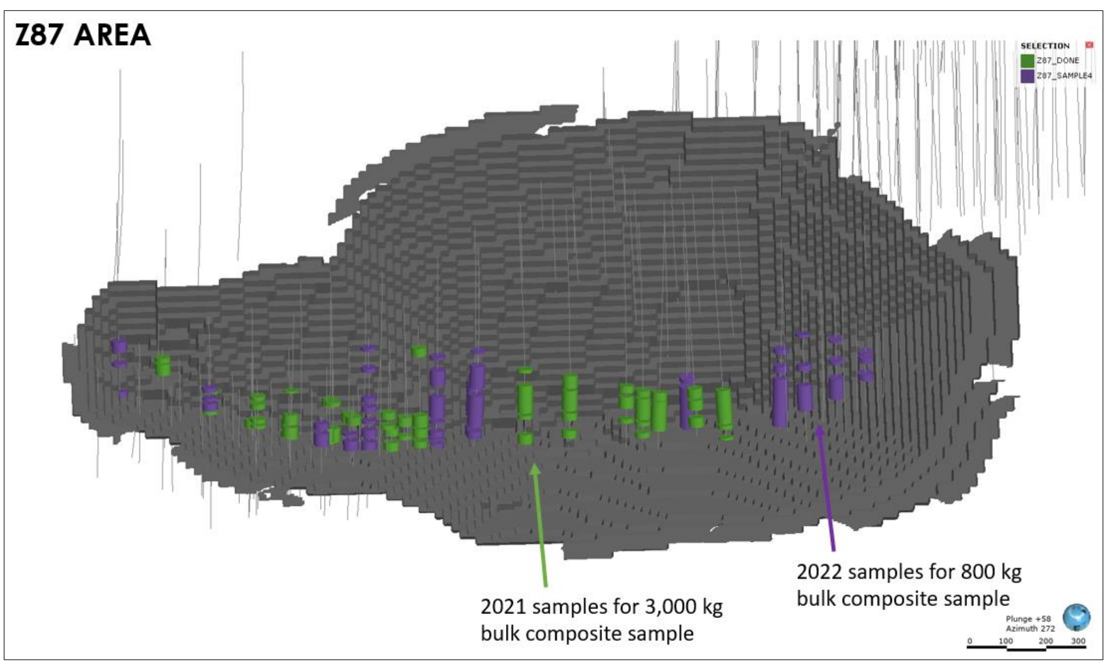
Figure 13-1 to Figure 13-5 show the location of the individual samples used for forming the bulk composite samples for the four zones.

Figure 13-1: Sample Location for J-Zone – 3,000 kg and 800 kg Samples



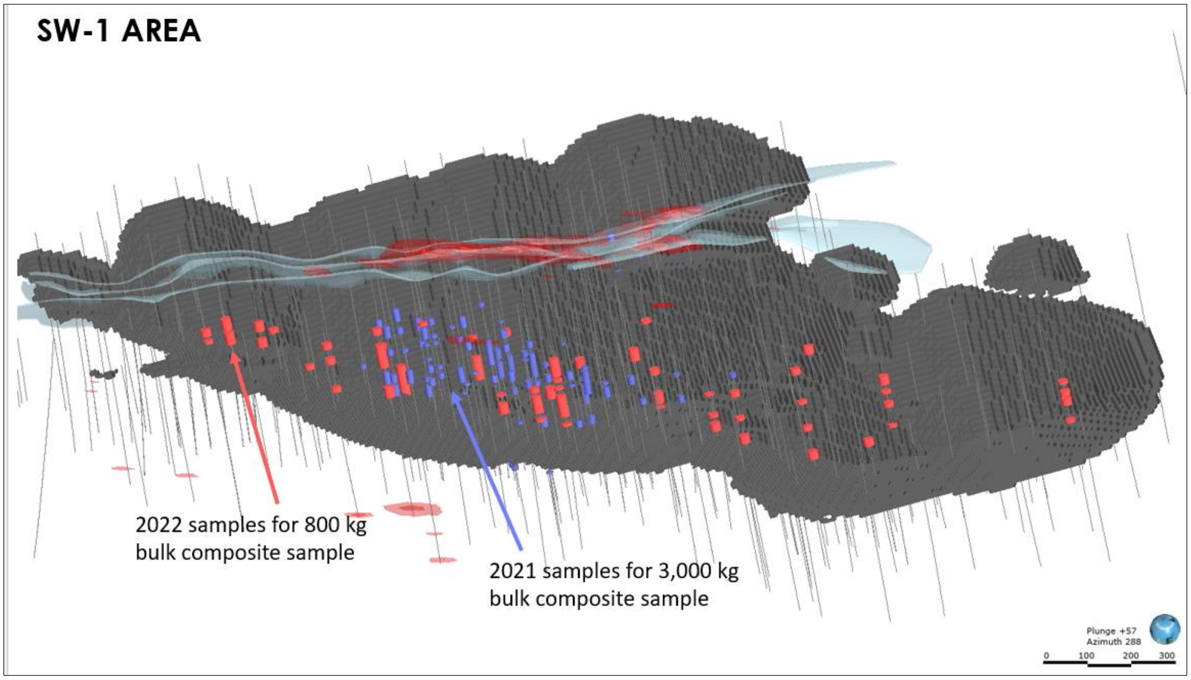
Source: Troilus (2023)

Figure 13-2: Sample Location for 87 Zone- 3,000 kg and 800 kg Samples



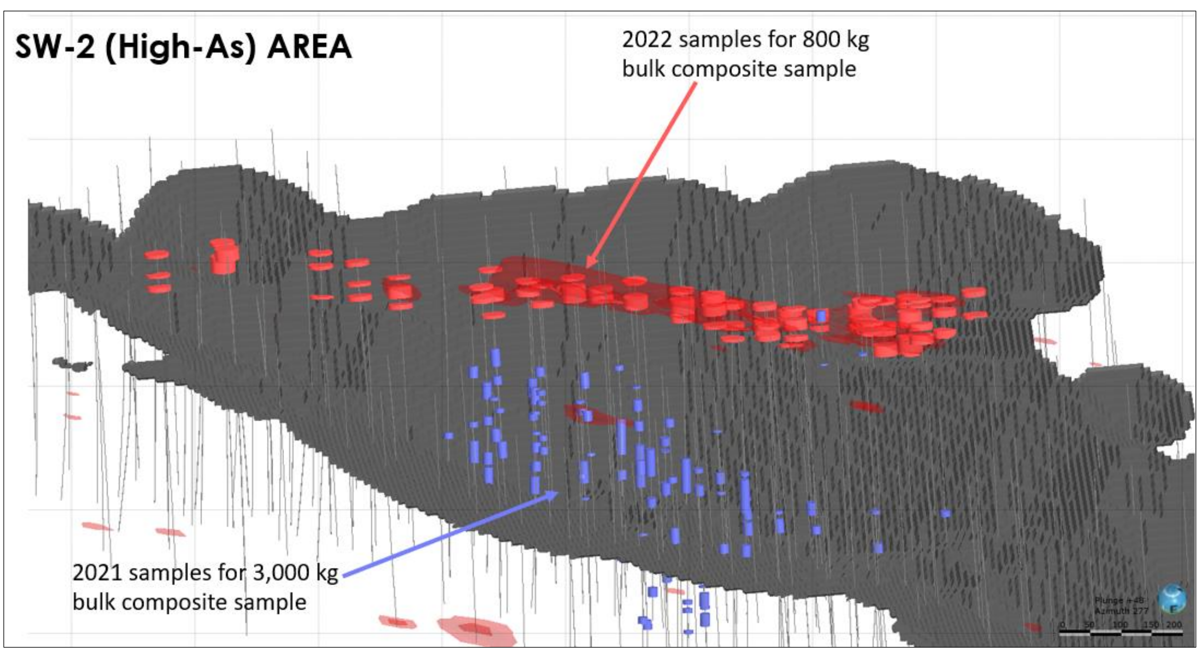
Source: Troilus (2023)

Figure 13-3: Sample Location for SW-1 – 3,000 kg and 800 kg samples



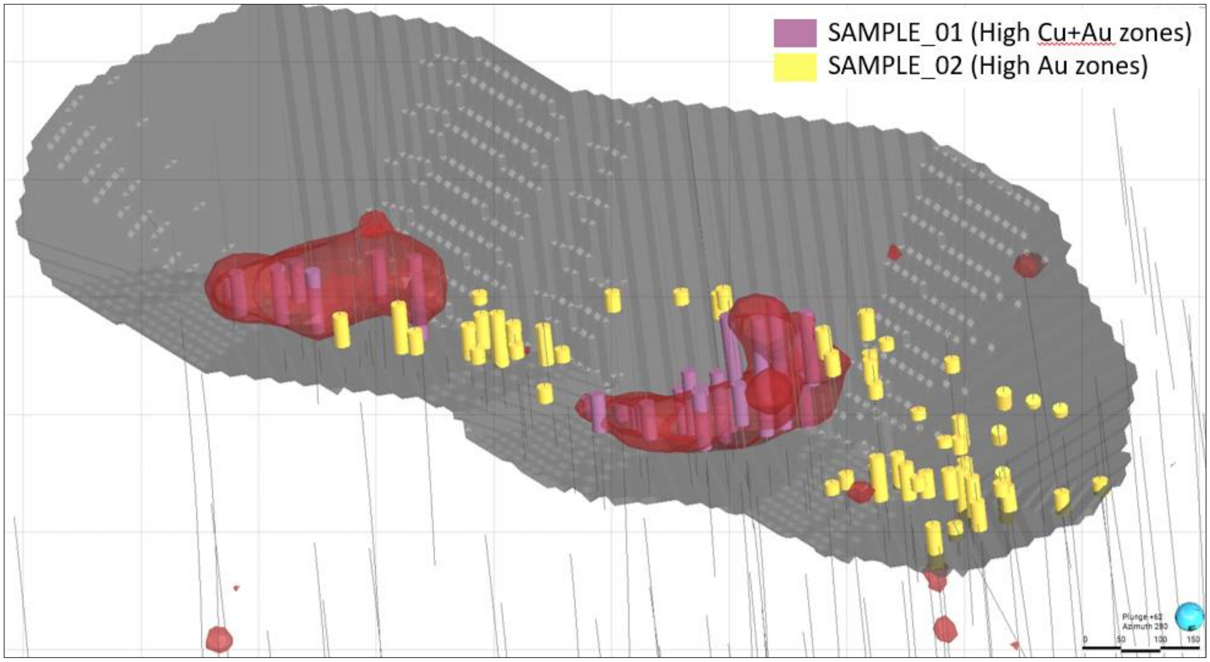
Source: Troilus (2023)

Figure 13-4: Sample Location for SW-2 - 3000, and 800 kg samples



Source: Troilus (2023)

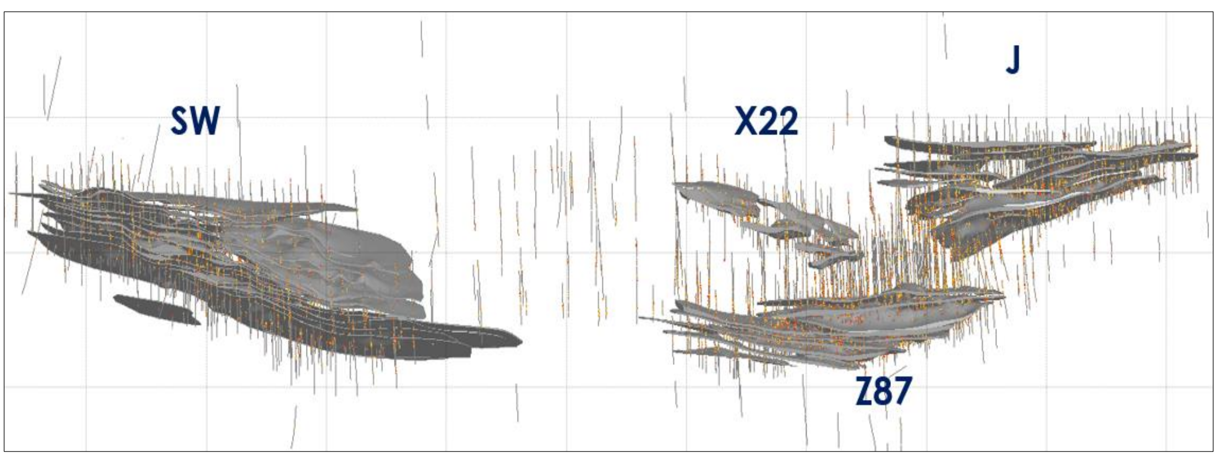
Figure 13-5: Sample Location for X22 Zone 1 and 2 – 1,500 kg Samples



Source: Troilus (2023)

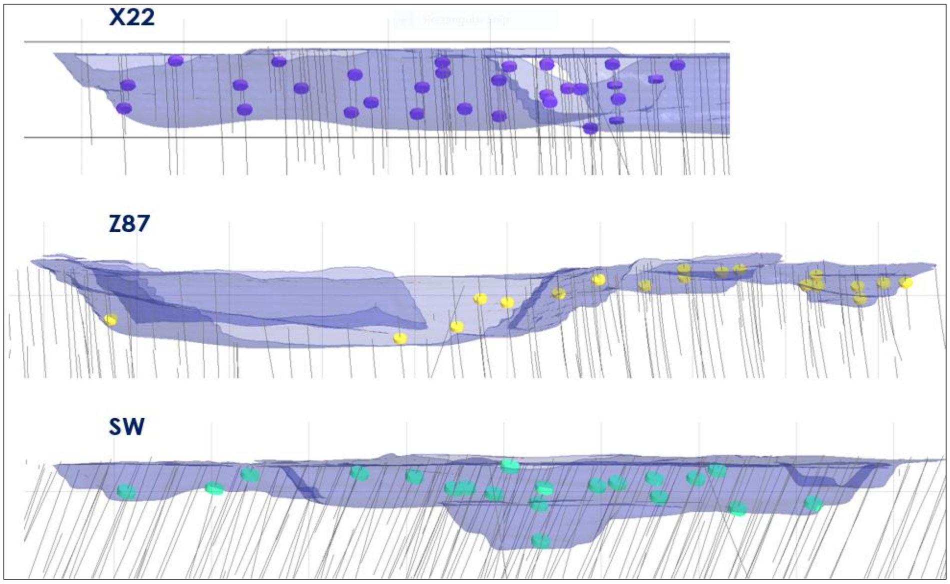
As part of the FS testwork program, a set of 20 samples were also selected for hardness variability testing. The samples covered different lithologies from the four zones, namely diorite, tonalite, intermediate volcanics, porphyritic felsic, magnetic brecia, and mafic volcanics. Another 2 composites samples were also selected for HPGR testing. The first HPGR sample represents Year 1 to 3 mill feed and the second sample represents the LOM mill feed. Refer to Figure 13-6 for spatial representation of the mineralized zones for which the hardness variability samples were collected. Refer to Figure 13-7 and Figure 13-8 for locations of the HPGR samples.

Figure 13-6: Mineralized Zones Located Spatially – View Looking West



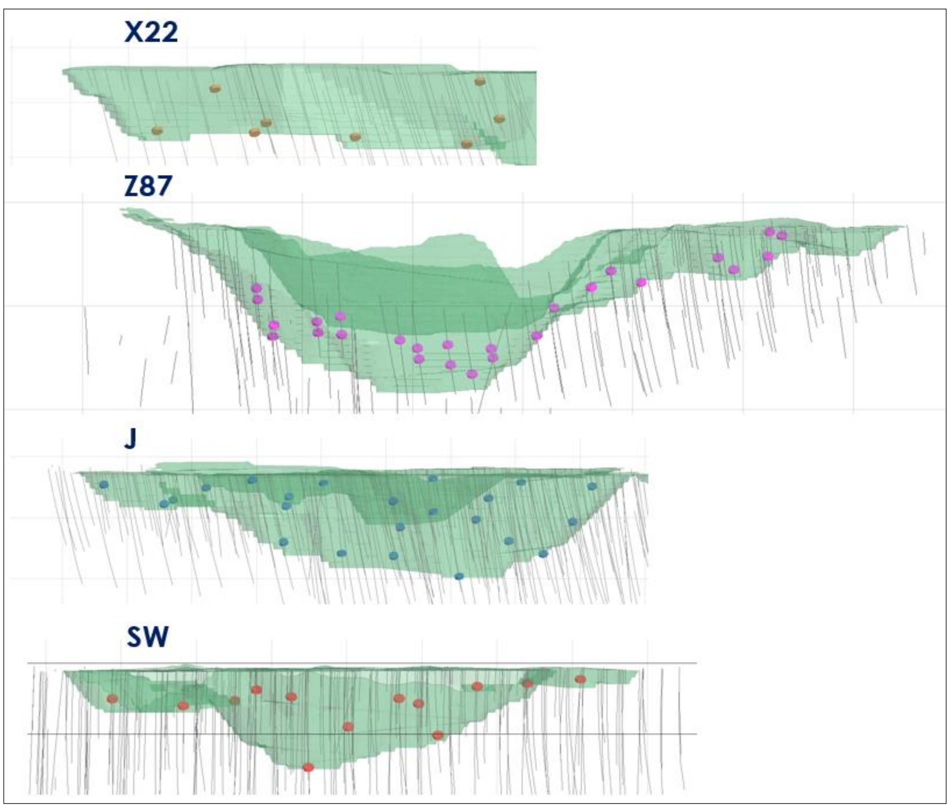
Source: Troilus (2023)

Figure 13-7: HPGR Pilot Plant Sample Selection – First 3 Years of Mill Feed Sample



Source: Troilus (2023)

Figure 13-8: HPGR Pilot Plant Sample Selection – Life-of-Mine Sample



Source: Troilus (2023)

13.4 Metallurgical Testwork Overview

Testwork results in the sections to follow will be presented in the following order:

- Comminution testwork:
  - SMC, abrasion and BW_i at JKTech in July 2021
  - SMC, abrasion and BW_i on variability samples at Base Metallurgical Laboratories (Base Met) in 2023
  - HPGR testwork at KCA in 2023 with review by J-Consultores Ltda.
  - SMC, abrasion and BW_i on HPGR samples at Hazen Research Inc. in 2023
  - ATWAL abrasion testwork at FLS in 2024
- Gravity separation testwork:
  - E-GRG testwork at FLS from 2021 to 2024
  - intensive cyanidation leaching on gravity concentrate at Base Met in 2021
  - gravity recovery predictions from FLS modelling & from historical Troilus data
- Pilot Plant Flotation testwork at Eriez from 2021 to 2024
- Supplementary testwork of samples produced during the Eriez pilot plant testwork by Base Met from 2021 to 2024 which includes:
  - mineralogical analysis on flotation feed and concentrate
  - cyanide leach testwork on cleaner tailings
  - settling testwork on flotation tailings
  - filtration testwork on the flotation concentrate
- Settling testwork at Pocock in 2020
- Acid base accounting testwork and geochemical characterization of samples from the Eriez pilot plant test with Troilus environmental team

13.5 Comminution Testwork

13.5.1 JKTech – SMC, Abrasion and Bond Mill Work Index Tests (2021 - 2022)

Three samples were submitted to JKTech via Base Met for SMC analysis in July 2021. A summary of the results is represented in Table 13-2.

Table 13-2: Comminution Test Results (JKTech – July 2021)

Sample ID	Bond Ball Work Index					RW_i (kWh/t)	A_i	SMC A x b
	Closing Screen (µm)	F_{80} (µm)	P_{80} (µm)	Gpb	BW_i (kWh/t)			
SW Domain	106	2,123	77	1.3	14.7	17.2	0.40	26.0
Zone 87	106	2,052	75	1.8	11.1	15.2	0.34	27.0
Zone J4/J5	106	1,869	79	1.9	11.2	13.7	0.20	30.0
Zone J4/J5	75	1,907	62	1.4	13.1			

Based on the SAG Mill Comminution (SMC) A x b values shown in Table 13-3, these samples are classified as hard when compared to other samples in the JKTech database. The Bond abrasion index of the samples was measured to be between 0.20 and 0.40, indicating that the samples are moderately abrasive. Based on BW_i results, the Z87 Zone and J4/J5 Zone samples are classified as moderately soft while the SW Zone sample are classified as moderately hard with respect to ball milling. Lastly, based on the RW_i results, all the samples are considered moderately hard to hard with respect to rod milling.

13.5.2 Base Met Lab – SMC, Abrasion and Bond Mill Work Index Tests (2023)

With the guidance from OMC, a set of 20 samples were selected and sent to Base Met for hardness variability testing. A summary of the hardness variability testwork is shown in Table 13-3.

Table 13-3: Hardness Variability Testwork (Base Met, 2023)

Samples	Zone	Lithology	Axb	BW_i, kWh/t	Dwi kWh/m^3	Mia, kWh/t	Mih, kWh/t	Mic, kWh/t	A_i, g
1	X22	Porphyritic Felsics	30.0	11.6	9.2	24.4	19.2	9.9	0.3
2	X22	Diorite	29.5	13.6	9.7	24.9	19.9	10.3	0.3
3	X22	Tonalite	29.0	13.0	9.4	25.3	20.1	10.4	0.4
4	X22	Tonalite	31.6	13.2	8.7	23.6	18.4	9.5	0.4
5	87	Porphyritic Felsics	27.1	11.1	10.0	26.7	21.4	11.1	0.3
6	87	Porphyritic Felsics	28.7	10.4	9.7	25.2	20.0	10.4	0.3
7	87	Basaltic Andesite	26.0	11.6	10.7	27.1	22.0	11.4	0.2
8	87	Basaltic Andesite	27.0	12.6	10.5	26.3	21.3	11.0	0.2
9	87	Intermediate Volcanics	35.3	9.8	7.7	21.4	16.3	8.4	0.2
10	87	Intermediate Volcanics	34.8	9.6	7.9	21.7	16.6	8.6	0.3
11	J	Porphyritic Felsics	30.0	12.5	8.9	24.6	19.3	10.0	0.4
12	J	Mafia Volcanics	30.0	13.1	9.9	24.3	19.4	10.0	0.2
13	J	Basaltic Andesite	32.0	10.2	9.5	23.1	18.3	9.5	0.3
14	J	Basaltic Andesite	31.0	11.7	8.8	23.6	18.4	9.5	0.3
15	J	Mafia Volcanics	30.0	10.4	10.0	24.3	19.5	10.1	0.3
16	J	Mafia Volcanics	34.5	11.2	8.2	21.7	16.7	8.6	0.2
17	SW	Porphyritic Felsics	25.7	16.2	10.6	28.0	22.7	11.8	0.4
18	SW	Magnetic Breccia	26.0	14.5	11.3	26.8	22.0	11.4	0.2
19	SW	Diorite	29.0	13.7	10.1	24.6	19.7	10.2	0.3
20	SW	Mafia Volcanics	24.0	18.0	12.5	28.5	23.9	12.4	0.4

Based on the results, all samples are considered as "hard" when plotted against the entire JKTech database.

13.5.3 KCA - HPGR Pilot Plant Testwork (2023)

High pressure grinding testwork was conducted on portions of crushed composite material using a pilot-scale HPGR (PILOTWAL unit) at Kappes, Cassiday & Associates (KCA) in Reno, Nevada. Two samples were submitted for testing, Composite 01 – first 3 years of mill feed and Composite 02 – life of mine composite.

The PILOTWAL unit located at KCA's facility is equipped with two 30 kW motors and two 0.5 m diameter by 0.3 m long studded rolls. The objectives in sizing HPGRs are to meet the throughput requirements and to achieve a certain product fineness. The key parameters are the specific throughput rate and the specific press force required to obtain the desired comminution result.

The test program conducted at KCA was designed to determine the influence of several parameters on the operation of the HPGR and the products generated. In the single pass tests, specific grinding force and feed moisture content were varied. Locked cycle testing was conducted to determine the crushing performance when operating in closed circuit with a 5.16 mm screen. A summary of the PILOTWAL HPGR testwork is presented in Table 13-4.

Table 13-4: Summary of PILOTWAL HPGR Tests

KCA Sample No.	Description	HPGR Testing	Test	HPGR Test No.	Test Feed, kg	Fresh Feed, kg	Screen Oversize	Initial Hydraulic Press., bar	Specific Force, N/mm²	Feed Moisture	Product Moisture
98601 A	Comp 01	Single Pass	S1	98603	120	120	--	65	2.10	3%	2.7%
			S2	98604	140	140	--	105	3.38	3%	2.9%
			S3	98605	120	120	--	135	4.35	3%	2.7%
			S4	98606	120	120	--	105	3.43	1%	1.4%
			S5	98607	120	120	--	105	3.40	5%	4.2%
98601 A	Comp 01	Locked Cycle (5.16 mm)	S2.1	98604	140	140	35%	105	3.38	3%	2.9%
			S2.2	98604	70	100	36%	105	3.35	3%	--
			S2.3	98604	70	100	35%	105	3.35	3%	3.1%
98602 A	Comp 02	Single Pass	S6	98608	120	120	--	65	2.12	3%	3.2%
			S7	98609	140	140	--	105	3.38	3%	3.2%
			S8	98610	120	120	--	135	4.39	3%	2.9%
			S9	98611	120	120	--	105	3.39	1%	1.1%
			S10	98612	120	120	--	105	3.35	5%	4.1%
98602 A	Comp 02	Locked Cycle (5.16 mm)	S7.1	98609	140	140	38%	105	3.38	3%	3.2%
			S7.2	98609	70	100	35%	105	3.32	3%	--
			S7.3	98609	70	100	35%	105	3.31	3%	2.9%

Note: Locked cycle product was wet screened for recycling.

Size distributions of the discharge were determined after the first and third cycles to identify any differences between open circuit and closed circuit PILOTWAL operation. The size distributions of the screen oversize and undersize after the third cycle were determined to analyze the screen performance and to get a measure on any pronounced fines carry over due to cake formation.

For both composites, the particle size distributions of the fully pressed center material and the total discharge were similar between the first and third cycles of the test. The screen analysis of the third cycle oversize material showed a screen bypass of less than 5%. These factors indicate competent cake formation was not a significant issue during testing.

In addition to the testwork conducted at KCA, portions of the composite samples were outsourced to FLSmidth Mining Technologies GmbH in Germany for ATWAL abrasion testwork, and to Hazen Research, Inc. for Bond Work Index and SMC testwork.

13.5.4 FLSmidth – High Pressure Grinding Abrasions Testwork (2024)

Two samples were sent via KCA to an FLS research center in Germany for ATWAL abrasion testwork. Four ATWAL tests were carried out on the samples and the results are summarized in Table 13-5.

Table 13-5 ATWAL Abrasion Testwork (FLS, 2024)

Sample	Test	Top Feed Size (mm)	Moisture (% H_2O)	Grinding Force (N/mm^2)	Wear Rate (g/t)
Comp 01 (98601A)	A1	3.15	1.0	4	20.9
Comp 01 (98601A)	A2	3.15	3.0	4	25.5
Comp 02 (98602A)	A3	3.15	1.0	4	19.7
Comp 02 (98602A)	A4	3.15	3.0	4	23.7

Based on the results in Table 13-5, the “Comp 01” sample was classified as “medium abrasive” at 1% and at 3 % moisture content. The “Comp 02” sample was classified as “low / medium abrasive” at 1 % moisture and as “medium abrasive” at 3 % moisture.

13.5.5 Hazen Research Inc. - SMC and Bond Mill Work Index Tests (2023)

HPGR samples were sent via KCA to Hazen Research Inc. for SMC, abrasion and BW_i testwork. Table 13-6 shows a summary of the results on the HPGR head samples and HPGR products. Samples 98604H, and 98609H were not coarse enough to perform SMC and abrasion tests and only BW_i tests were completed for them. Compared to the ore samples, the HPGR product had a small reduction in BW_i after being processed through the HPGR. OMC opted to be conservative and used the pre-HPGR result for comminution design.

Table 13-6: HPGR Product Comminution Test Results

Hazen ID	Client ID	Sample Description	BW_i, kWh/t	A_i, g	SMC (A x b)
56011-1	98601 (Comp 1)	Head Sample	11.7	0.3721	27.3
56011-2	98602 (Comp 2)	Head Sample	10.9	0.2633	29.3
56011-3	98604 H	HPGR Product	11.0	N/A	N/A
56011-4	98609 H	HPGR Product	10.2	N/A	N/A

13.5.6 Third Party Review of HPGR Circuit by J-Consultores Ltda. (2023)

Troilus contracted J-Consultores Ltda., (JCL) to assist with the project as a third-party reviewer for the comminution circuit design and testing program. Table 13-7 shows the pilot testwork results reviewed by JCL. Preliminary trials included 14 tests with 2 different composites, under various specific grinding forces.

The findings were as follow:

- The HPGR testwork results allowed for the estimation of all intrinsic parameters so that the HPGR model could be well calibrated against empirical data.
- Locked cycle test results were not included in the analysis.
- M-dot was found to be in the range of 290 to 305 (t·s)/(m^3·h), quite typical for this type of application, when operating in open circuit mode.
- For a design capacity of 50 ktpd, at 88% run-time, a 3.0 m x 2.0 m HPGR unit would serve well the purpose, always operating below the recommended maximum of 21 rpm.
- Circulating loads were found to be within typical industry ranges, decreasing as screen opening increases.

Table 13-7: Pilot HPGR Testwork Results

Sample	Comp 01	Comp 01	Comp 01	Comp 01	Comp 01	Comp 01	Comp 01	Comp 02	Comp 02	Comp 02	Comp 02	Comp 02	Comp 02	Comp 02
Test N°	S1 98603	S2.1 98604	S3 98605	S4 98606	S5 98607	S2.2 98604	S2.3 98604	S6 98608	S7.1 98609	S8 98610	S9 98611	S10 98612	S7.2 98609	S7.3 98609
Ore Hardness (Axb, SPI or BWI)														
Ore Density, t/m3	2.68	2.68	2.68	2.68	2.68	2.68	2.68	2.68	2.68	2.68	2.68	2.68	2.68	2.68
Roll Diameter, D, m	0.500	0.500	0.500	0.500	0.500	0.500	0.500	0.500	0.500	0.500	0.500	0.500	0.500	0.500
Roll Length, L, m	0.300	0.300	0.300	0.300	0.300	0.300	0.300	0.300	0.300	0.300	0.300	0.300	0.300	0.300
Peripheral Speed, m/s	0.200	0.200	0.200	0.200	0.200	0.200	0.200	0.200	0.200	0.200	0.200	0.200	0.200	0.200
Rotational Speed	7.64	7.64	7.64	7.64	7.64	7.64	7.64	7.64	7.64	7.64	7.64	7.64	7.64	7.64
Sp. Grinding Force, Newton/mm2	2.10	3.38	4.35	3.43	3.40	3.35	3.35	2.12	3.38	4.39	3.39	3.35	3.32	3.31
Compression Force, kNewton	314	506	653	515	510	502	503	318	507	658	508	502	497	496
Friction Coefficient, μ	0.0738	0.0677	0.0682	0.0687	0.0683	0.0658	0.0642	0.0752	0.0691	0.0688	0.0695	0.0695	0.0651	0.0658
Net Power Draw, kW	9.29	13.72	17.80	14.16	13.92	13.21	12.92	9.56	14.02	18.11	14.14	13.96	12.95	13.07
Critical Compression Angle, α_c	8.07	7.55	7.38	7.64	7.60	7.20	7.24	8.06	7.52	7.26	7.64	7.60	7.17	7.20
Operational Gap, mm	15.43	13.60	13.01	13.89	13.76	12.40	12.53	15.39	13.47	12.60	13.89	13.75	12.29	12.40
Extrusion/Slippage Factor	1.16	1.28	1.33	1.29	1.27	1.28	1.27	1.16	1.29	1.36	1.29	1.29	1.33	1.32
HRC Grinding Capacity, t/h (wet)	9.06	8.83	8.73	8.95	8.96	8.08	8.09	9.09	8.83	8.69	8.90	9.08	8.31	8.27
Feed Moisture, %	2.68	2.86	2.66	1.36	4.16	3.09	3.09	3.23	3.22	2.88	1.09	4.11	2.93	2.93
HRC Grinding Capacity, t/h (dry)	8.82	8.58	8.50	8.83	8.58	7.83	7.84	8.80	8.55	8.44	8.80	8.70	8.06	8.02
Mdot (wet)	**302.0**	**294.5**	**290.9**	**298.3**	**298.5**	**269.3**	**269.8**	**303.1**	**294.4**	**289.7**	**296.7**	**302.6**	**276.9**	**275.6**
Specific Energy, kWh/t	1.05	1.60	2.10	1.60	1.62	1.69	1.65	1.09	1.64	2.15	1.61	1.60	1.61	1.63
Ore Density, rs, ton/m3	2.68	2.68	2.68	2.68	2.68	2.68	2.68	2.68	2.68	2.68	2.68	2.68	2.68	2.68
Bulk Voids, %	35.00	35.00	35.00	35.00	35.00	35.00	35.00	35.00	35.00	35.00	35.00	35.00	35.00	35.00
Cake Voids, %	15.00	15.00	15.00	15.00	15.00	15.00	15.00	15.00	15.00	15.00	15.00	15.00	15.00	15.00
App. Density, rap, ton/m3	1.790	1.793	1.790	1.766	1.818	1.798	1.798	1.800	1.800	1.794	1.761	1.817	1.795	1.795
App. Density, rc, ton/m3	2.341	2.345	2.340	2.309	2.377	2.351	2.351	2.354	2.354	2.346	2.303	2.376	2.347	2.347
Screen Opening, mm	5.16	5.16	5.16	5.16	5.16	5.16	5.16	5.16	5.16	5.16	5.16	5.16	5.16	5.16
O'Size, % of Feed	31.59	34.71	23.86	21.47	25.16	35.89	34.60	31.25	37.77	24.78	29.75	31.84	35.41	35.29
U'Size, % of Feed	68.41	65.29	76.14	78.53	74.84	64.11	65.40	68.75	62.23	75.22	70.25	68.16	64.59	64.71
Circ. Load Ratio	0.462	0.532	0.313	0.273	0.336	0.560	0.529	0.455	0.607	0.329	0.424	0.467	0.548	0.545
SELECTION FUNCTION (SIMPLIFIED MODEL)														
alpha0	0.01024	0.00821	0.00714	0.00963	0.00889		0.00808	0.01147	0.00980	0.00803	0.00869	0.00823		0.00985
alpha1	0.499	0.499	0.499	0.499	0.499		0.499	0.489	0.489	0.489	0.489	0.489		0.489
alpha2	1.939	1.939	1.939	1.939	1.939		1.939	2.314	2.314	2.314	2.314	2.314		2.314
Dcrit	14000	14000	14000	14000	14000		14000	14000	14000	14000	14000	14000		14000

13.6 Gravity Separation

13.6.1 FLSmidth Gravity Recovery Testwork (2021 - 2022)

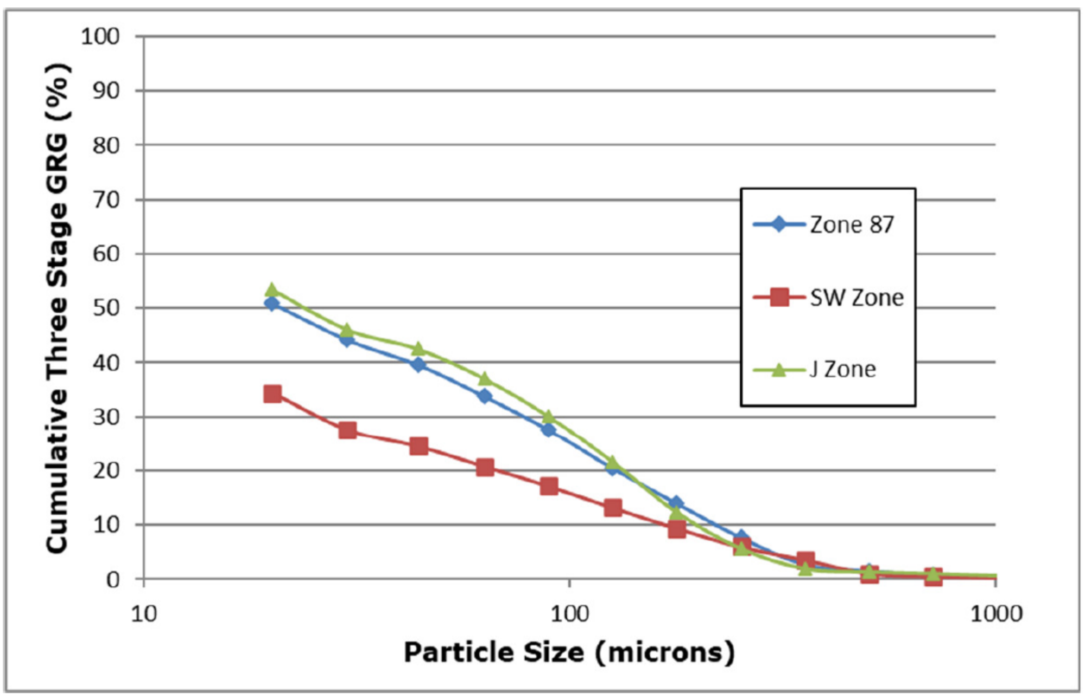
FLSmidth received three samples from Eriez for conducting the E-GRG testwork for J, SW, and 87 zones. The E-GRG testwork is based on progressive particle size reduction to allow recovery of gold while minimizing over grinding. The results are reported for the overall gold recovery and E-GRG value as a function of particle size distribution.

The overall gravity recovery results for each zone are presented in Table 13-8 and Figure 13-9. The as tested samples were ground to a final test grind size different to that of the actual plant grind size of 75 µm, hence, a correction was done to predict the GRG at the plant cyclone overflow P_{80}.

Table 13-8 GRG Values for J-Zone, SW-Zone, and Zone 87 (FLS, 2021 - 2022)

Ore Sample	Corrected GRG (%)	Test P_{80} (µm)	As Tested GRG (%)
J-Zone	53.5	73.0	54.8
SW-Zone	34.4	80.0	34.6
Zone 87	51.0	103.0	47.6

Figure 13-9 GRG Particle Size Distribution for J-Zone, SW-Zone, and Zone 87 (FLS, 2021 - 2022)



Source: FLS (2022)

All the concentrates from the three zones were rated on the Amira scale to be between moderate to coarse.

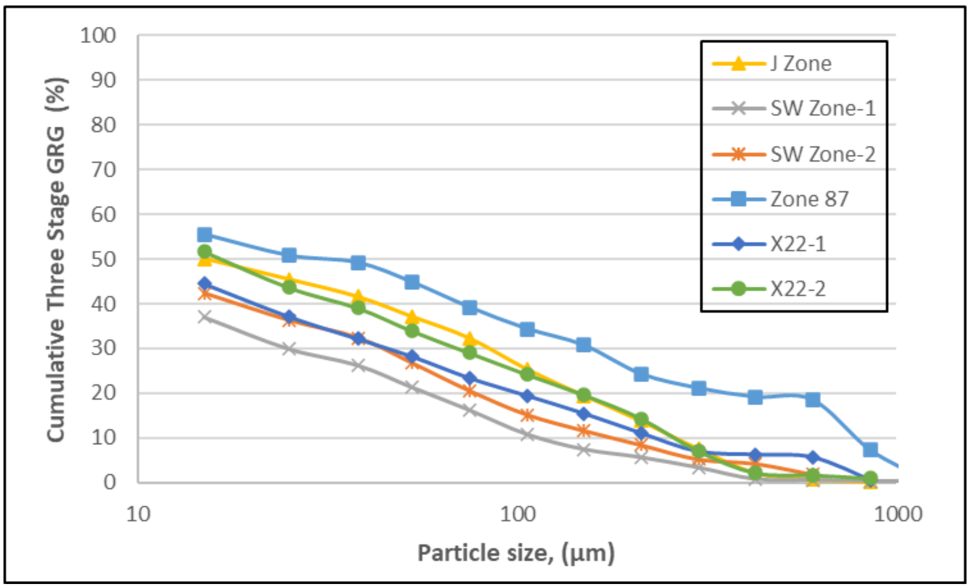
13.6.2 FLSmidth Gravity Recovery Testwork (2023 - 2024)

FLSmidth received six samples from Eriez for conducting the E-GRG testwork for J, SW1, SW2, 87, X22-1, and X22-2 zones. The results are shown in Table 13-9 and Figure 13-10. J and SW samples were over-crushed to -20 mesh by Eriez (-10 mesh is the standard procedure), however, the impact on GRG recovery was insignificant.

Table 13-9: GRG Values for J, SW1, SW2, 87, X22-1 & X22-2 (FLS, 2023 – 2024)

Ore Sample	Test P_{80} (µm)	GRG Value (%)
J-Zone	73	50.1
SW-Zone 1	73	37.1
SW-Zone 2	63	42.3
Zone 87	73	55.5
X22-1	77	44.5
X22-2	81	51.7

Figure 13-10 GRG Particle Size Distribution for J, SW1, SW2, 87, X22-1 and X22-2 (FLS, 2023 – 2024)



Source: Lycopodium (2024)

Gravity concentrates from all the zones were rated on the Amira scale to be between moderate to coarse/very coarse. The testwork on the new samples show that the GRG recoveries were consistent, and similar to the earlier E-GRG test results across the zones. However, the zone 87 sample produced a significantly coarser GRG distribution compared to the earlier E-GRG test result.

13.6.3 Base Metallurgical Laboratory - Intensive Leach Tests on Knelson Concentrates

Intensive leach test was performed by Base Met Lab on three gravity concentrates; however, the results were never used as Troilus opted for a cyanide-free operation and therefore the gravity concentration product will be further treated using shaking tables instead of intensive cyanidation leaching.

13.6.4 Gold Gravity Recovery – FLS Modelling and Data from Historical Troilus Operation

FLSmidth provided gravity recovery modelling for the six ore zones (J, SW-1, SW-1, 87, X22-1, X22-2) based on the E-GRG testwork conducted from 2021 - 2023. The modelling was conducted based on gravity recovery in the primary milling circuit and regrind circuit using Knelson centrifugal concentrators and shaking tables.

Gravity recovery modelling results for the primary milling circuit are shown in Table 13-10.

Table 13-10: Gravity Recovery Modelling Results for Primary Milling Circuit

Ore Zone	Solids t/h Treated	No. of Units	% Circulating Load Treated	Location of Feed to Gravity	GRG Recovery %	Gold Gravity Recovery %	Conc Mass (kg)
J	1,600	2 x QS70	18.8	Cyclone U/F	54	28	9,000
SW-1	1,600	2 x Q270	18.8	Cyclone U/F	47	16	9,000
SW-2	1,600	2 x Q270	18.8	Cyclone U/F	52	21	9,000
87	1,600	2 x Q270	18.8	Cyclone U/F	57	27	9,000
X22-1	1,600	2 x QS70	18.8	Cyclone U/F	48	19	9,000
X22-1	1,600	2 x QS70	18.8	Cyclone U/F	51	28	9,000

Gravity recovery modelling results for the regrind circuit are shown in Table 13-11. The prediction from FLSmidth has been based on an open regrind circuit.

Table 13-11: Gravity Recovery Modelling Results for Regrind Circuit

Ore Zone	Solids t/h Treated	No. of Units	Gold Gravity Recovery %
J	236.7	2 x QS48	1.3
SW-1	236.7	2 x QS48	1.4
SW-2	236.7	2 x QS48	1.5
87	236.7	2 x QS48	1.3
X22-1	236.7	2 x QS48	1.7
X22-2	236.7	2 x QS48	2.0

Note that FLS modelling for the regrind circuit only considered the GRG that can exit through the cyclone overflow from the primary milling circuit to feed the regrind circuit. The modelling has not accounted for the portion of GRG that will be liberated when grinding from 75 µm to 20 µm as it is not possible to know how much of this “yet to be liberated” GRG would have made it to the flotation concentrate. As a result, the regrind gold gravity recoveries reported in Table 13-12 have understated the potential gravity recovery in the regrind circuit.

Table 13-12 presents the gold gravity recovery reported for the historical Troilus operation where shaking tables were also used then.

Table 13-12: Gravity Recovery for Historical Troilus Operation

Year	Gold Gravity Recovery%
2000	28.1
2001	26.7
2002	28.2
2003	36.2
2004	35.2
2005	34.5
2006	34.7
2007	34.9
2008	30.6
2009	34.7
2010	32.0
Average	**32.3**

The historical data for Troilus historical operation indicates that gold gravity recovery on average was 32% from year 2000 to 2010.

13.7 Flotation Testwork

Troilus retained Eriez to perform bench scale and pilot plant testwork using the three bulk composite 3,000 kg samples from J, SW, and 87 for the test program in 2021. Further testing was later completed on additional 800 kg and 1,500 kg samples from the J, SW, 87 zones, and from X22 zone, respectively, when the resource expanded during the FS phase. Two small samples for X22 (at 150 kg each) were also submitted to Eriez for preliminary flotation testwork.

The objective of the Eriez pilot program was to maximize metal recovery performance by defining optimum operating conditions (grind size, reagent additions and operating pH and oxidation potential) for use in processing ore from each of the zones. Additional objectives included the production of sufficient final concentrate to allow chemical characterization and the production of other intermediate products for additional testing by Base Met, which is discussed in Section 13.8. Due to the low copper grades of the deposits, large feed samples were necessary to ensure an adequate amount of material was available for closed circuit cleaner operation.

Table 13-13 presents the details of pilot testwork conducted at Eriez.

Table 13-13 Eriez Pilot Plant Testwork

Sample Zone	Mass	Milling Study R/S	Bulk Milling	Gravity	Column R/S	Column R/S Conc Milling Study	Column Cleaning	Open Circuit	Closed Circuit
J	3,000 kg	✓	✓	✓	✓	✓	✓	✓	✓
J	800 kg	✓	✓		✓	✓		✓	
SW	3,000 kg	✓	✓	✓	✓	✓	✓	✓	✓
SW	800 kg	✓	✓		✓	✓		✓	
SW (As)	800 kg	✓	✓		✓	✓		✓	
87	3,000 kg	✓	✓	✓	✓	✓	✓	✓	✓
87	800 kg	✓	✓		✓	✓		✓	
X22-1	1,500 kg	✓	✓		✓	✓	✓	✓	✓
X22-1	150 kg	✓	✓		✓	✓		✓	
X22-2	1,500 kg	✓	✓		✓	✓	✓	✓	✓
X22-2	150 kg	✓	✓		✓	✓		✓	

The Eriez testwork was conducted using a combination of bench scale tests with mechanical cells to establish the optimal flotation conditions followed by pilot plant column operation using closed-circuit milling to produce a rougher/scavenger concentrate. A gravity circuit was included only for the 3,000 kg samples for J, SW, and 87 zones. Refer to Figure 13-11, Figure 13-12, and Figure 13-13 for details. This phase of the testwork produced a Knelson concentrate (for only the 3,000 kg samples), R/S concentrate, and scavenger tailings.

Figure 13-11 Eriez Pilot Plant Ball Mill and Gravity Recovery Circuit

Source: Eriez (2023)

Figure 13-12 Eriez Pilot Plant 4-inch Rougher Column and 3-inch Scavenger Column

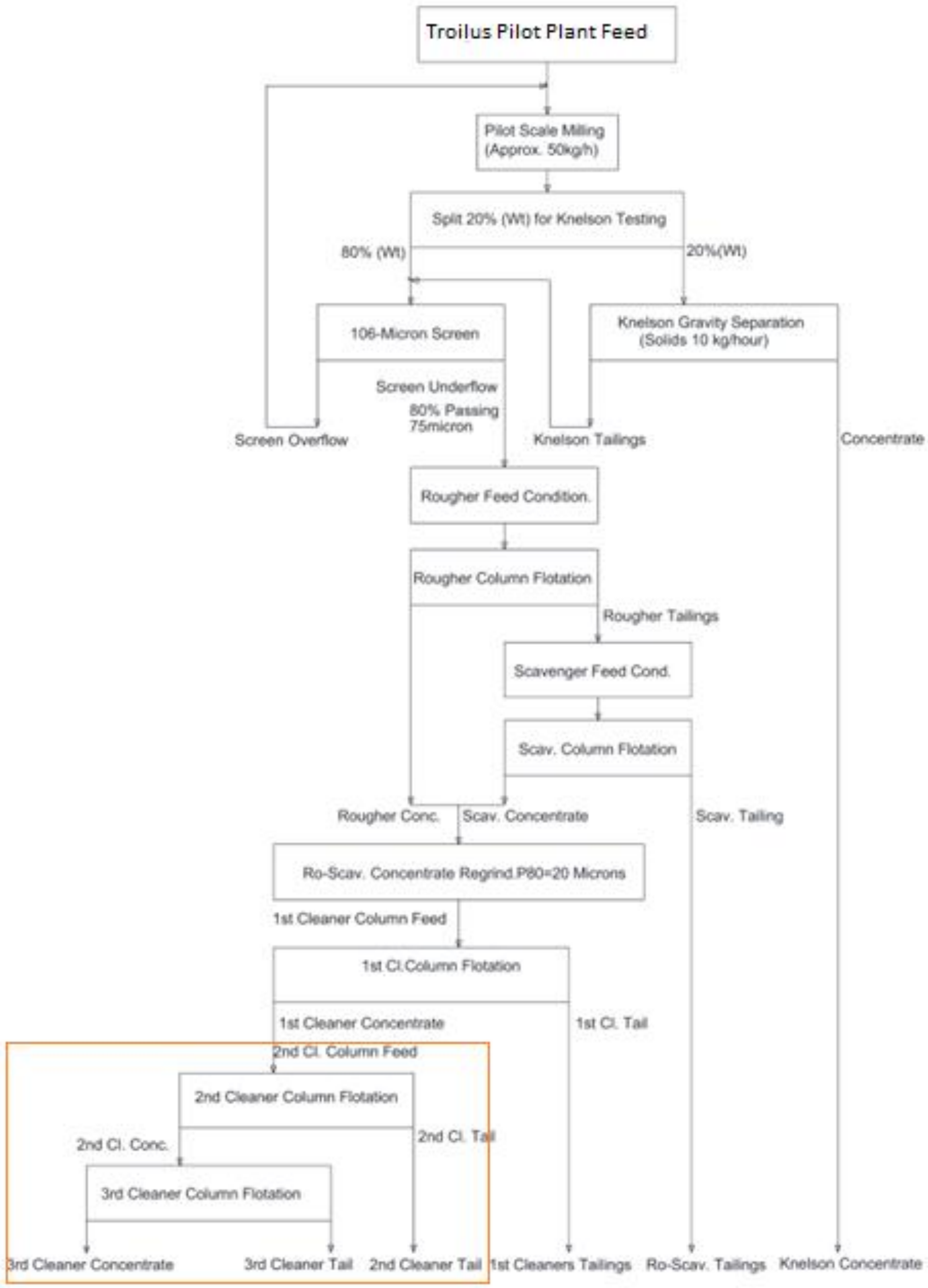
Source: Eriez (2023)

Figure 13-13 Eriez Pilot Plant 4-inch 1st Cleaner, 3-inch 2nd Cleaner and 2-inch 3rd Cleaner Columns

Source: Eriez (2023)

For the testwork corresponding to the 3,000 kg samples, a schematic flowsheet of the milling, gravity and classification circuit, R/S and cleaner circuit is shown in Figure 13-14. In this schematic, the cleaner circuit is in an open circuit configuration for the J-Zone sample and closed circuit for the SW and 87 samples as shown in the box.

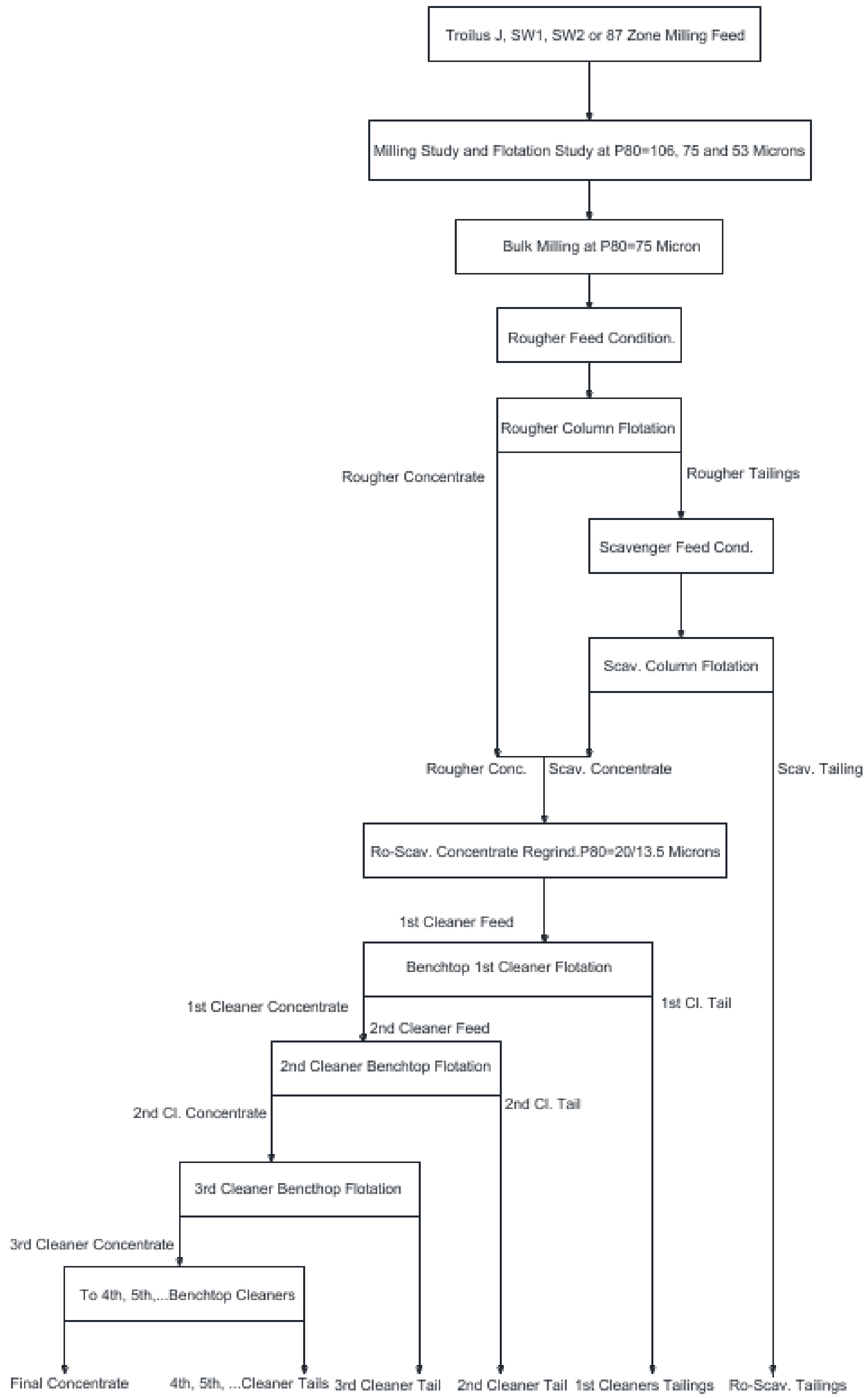
Figure 13-14 Schematic of Pilot Plant for J, SW, and 87 Zone 3,000 kg Samples



Source: Eriez (2023)

For the testwork corresponding to the 800 kg samples, a schematic of the milling, R/S and cleaner circuit is shown in Figure 13-15.

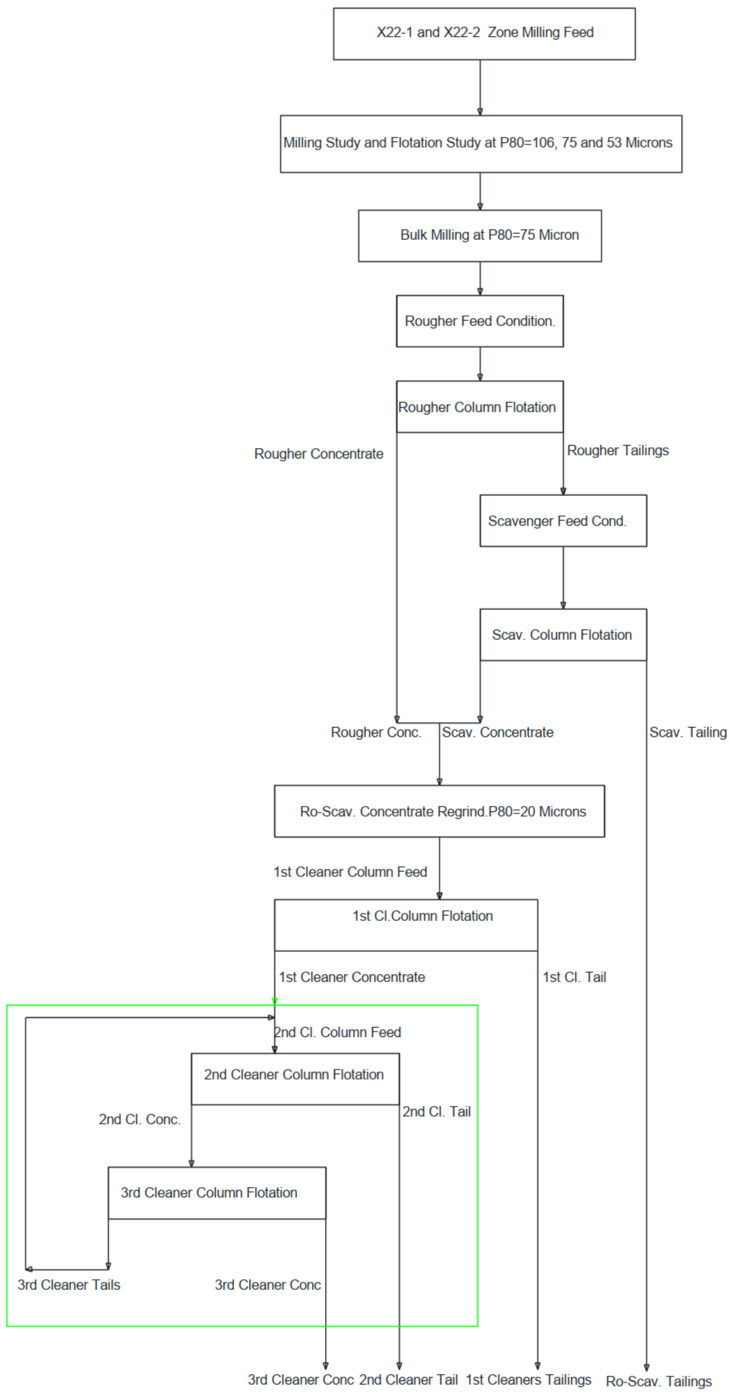
Figure 13-15 Testwork Schematic for J, SW1, SW2, 87 Zone 800 kg Samples



Source: Eriez (2023)

For the testwork corresponding to the 1,500 kg samples for X22, a schematic of the milling, R/S and cleaner circuit is shown in Figure 13-16.

Figure 13-16 Testwork Schematic for X22 Zone 1,500 kg Samples



Source: Eriez (2023)

The Eriez pilot plant testwork results are presented in the following sections.

13.7.1 Eriez Testwork Results for J, SW and 87 Zone (3,000 kg Samples)

Summaries of the Eriez pilot plant testwork results for the 3,000 kg samples are presented in Table 13-14, Table 13-15, and Table 13-16 for J, SW and 87 zones, respectively.

Table 13-14: J-Zone Knelson & Pilot Column Rougher-Scavenger-Cleaner Flotation Results

Streams	Mass Distribution (%)	Assays				Distribution (%)		
		Cu (%)	Fe (%)	Au (mg/kg)	Ag (mg/kg)	Cu	Au	Ag
As-Received Milled Feed	100	0.06	3.38	0.50	0.85	100.00	100.00	100.00
Knelson Feed	20.00	0.06	3.38	0.50	0.85	20.00	20.00	20.00
Knelson Conc.	0.18	0.16	9.48	36.74	9.69	0.48	13.23	2.05
Knelson Tail	19.82	0.06	3.32	0.17	0.77	19.52	6.77	17.95
Cal. Knelson Untreated Ro Feed	80.00	0.060	3.38	0.50	0.852	80.00	80.00	80.00
Column Rougher Feed	99.82	0.06	3.36	0.43	0.84			
Column Ro Conc	3.47	1.59	23.00	11.15	21.44	92.66	77.51	87.43
Column Ro Tail	96.35	0.00	2.66	0.05	0.09			
Column Scav Conc	2.23	0.11	5.47	0.87	2.09	4.29	3.89	5.48
Column Scav Tail	94.12	0.00	2.59	0.03	0.05	2.57	5.36	5.04
Comb. Knelson Conc and Col. Conc	5.88			8.04		97.43	94.63	94.96
Combined Column Ro-Scav. Conc.	5.70	1.01	16.14	7.13	13.87	96.94	81.40	92.91
Cleaners Column Flotation								
1st Cl. Feed	5.70	1.01	16.14	7.13	13.87	96.94	81.40	92.91
1st Cl. Tail	4.67	0.01	11.91	0.06	0.19	0.84	0.51	1.03
1st Cl. Conc	1.03	5.54	35.26	39.13	75.75	96.11	1.10	91.88
2nd Cl. Tail	0.37	0.36	25.74	0.37	4.39	2.20	2.82	1.90
2nd Cl. Conc	0.66	8.43	40.56	60.70	115.48	93.90	7.97	89.97
3rd Cl. Tail	0.27	0.87	37.44	1.69	10.21	3.95	2.99	3.23
3rd Cl. Conc	0.39	13.59	42.70	101.00	187.35	89.96	79.71	86.75
Comb. Knelson Conc & Column Conc	0.57	9.38	32.29	80.86	131.67	90.44	92.94	88.80

In terms of the flotation performance, the pilot test was a success for the J-Zone, resulting in an overall recovery to final concentrate (based on the as-received milled feed) of 90.44%, 92.94%, and 88.80% was achieved for copper, gold, and silver, respectively. The combined rougher-scavenger and cleaner circuit copper, gold, and silver flotation recoveries (based on the as-received milled Knelson feed) were approximately 89.96% at a copper grade of 13.6%, 79.71% at a gold grade of 101.0 g/t, and 86.75% at a silver grade of 187.4 g/t, respectively. The combined J-Zone weighted average of the flotation tails was calculated to be 0.006% Cu, and 0.035 g Au/t, 0.096 g Ag/t. The combined rougher-scavenger and cleaner KAX 51 collector, SPRI 206 collector and Na_2SO_3 depressant dosages, based on rougher feed, were approximately 59 g/t, 33 g/t and 300 g/t, respectively.

Table 13-15: SW-Zone Knelson & Pilot Column Rougher-Scavenger-Cleaner Flotation Results

Streams	Mass Distribution (%)	Assays				Distribution (%)		
		Cu (%)	Fe (%)	Au (mg/kg)	Ag (mg/kg)	Cu	Au	Ag
Knelson Feed (Head)	20.00	0.07	3.93	0.66	1.16	20	20	20
Knelson Conc.	0.20	0.18	18.64	20.77	21.97	0.56	6.22	3.75
Knelson Tail	19.80	0.0	3.79	0.46	0.96	19.44	13.78	16.25
Cal. Knelson Untreated Ro Feed	80.00	0.07	3.93	0.66	1.16	80.00	80.00	80.00
Column Rougher Feed	99.80	0.07	3.90	0.62	1.12			
Column Ro Conc	3.73	1.62	13.76	14.75	25.37	91.76	82.84	81.25
Column Ro Tail	96.07	0.01	3.52	0.08	0.18			
Column Scav Conc	1.00	0.32	4.85	2.71	8.13	4.89	4.07	6.96
Column Scav Tail	95.08	0.002	3.51	0.05	0.10	2.79	6.87	8.04
Comb. Knelson Conc and Col. Conc	4.92			12.55		97.21	93.13	91.96
Combined Column Ro-Scav. Conc.	4.73	1.34	11.88	12.21	21.7	96.66	86.91	88.21
Cleaners Column Flotation								
1st Cl. Feed	4.73	1.34	11.88	12.21	21.73	96.66	86.91	88.21
1st Cl. Conc	1.09	5.80	21.57	52.56	94.10	96.10	86.13	87.92
1st Cl. Tail	3.64	0.01	8.98	0.14	0.09	0.55	0.03	0.28
2nd Cl. Tail	0.69	0.2	13.62	3.92	3.76	2.62	2.39	2.23
3rd Cl. Conc	0.40	15.47	35.43	137.24	251.35	93.48	82.05	85.69
Comb. Knelson Conc & Column Conc	0.60	10.37	29.82	98.38	174.81	94.04	88.27	89.44
Combined Final Tail	99.40	0.004	3.78	0.08	0.12	5.96	11.73	10.56

Similar to the J-zone sample, the SW-zone performed well during the pilot test. An overall recovery (based on the as-received milled feed) of 94.0%, 88.3%, and 89.4% was achieved for copper, gold, and silver, respectively. The combined rougher-scavenger and cleaner circuit copper, gold, and silver flotation recoveries (based on the as-received milled Knelson feed) were approximately 93.5% at a copper grade of 15.5%, 82.1% at a gold grade of 137.2 g/t, and 85.7% at a silver grade of 251.4 g/t, respectively. The combined SW-Zone weighted average of the flotation tails was calculated to be 0.004% Cu, and 0.08 g Au/t, 0.12 g Ag/t. The combined rougher-scavenger and cleaner KAX 51 collector, SPRI 206 collector and Na_2SO_3 depressant dosages, based on rougher feed, were approximately 60.3 g/t, 40.7 g/t and 200 g/t, respectively.

Table 13-16: Zone 87 Knelson & Pilot Column Rougher-Scavenger-Cleaner Flotation Results

Streams	Mass Distribution (%)	Assays				Distribution (%)		
		Cu (%)	Fe (%)	Au (mg/kg)	Ag (mg/kg)	Cu	Au	Ag
Knelson Feed (Head)	20.00	0.070	3.235	0.643	0.298	20.00	20.00	20.00
Knelson Conc.	0.12	0.429	16.234	44.767	29.767	0.73	8.38	12.03
Knelson Tail	19.88	0.068	3.156	0.376	0.119	19.27	11.62	7.97
Cal. Knelson Untreated Ro Feed	80.00	0.070	3.235	0.643	0.298	80.00	80.00	80.00
Column Rougher Feed	99.88	0.070	3.219	0.590	0.262			
Column Ro Conc	2.35	2.826	19.980	24.333	10.504	94.59	89.06	83.02
Column Ro Tail	97.53	0.003	2.814	0.017	0.015			
Column Scav Conc	0.56	0.377	5.029	1.496	0.114	3.00	1.30	0.21
Column Scav Tail	96.97	0.001	2.802	0.008	0.015	1.67	1.27	4.73
Comb. Knelson Conc and Col. Conc	3.03			20.933		98.33	98.73	95.27
Combined Column Ro-Scav. Conc.	2.91	2.356	17.109	19.948	8.509	97.59	90.36	83.24
Cleaners Column Flotation								
1st Cl. Feed	2.91	2.356	17.109	19.948	8.509	97.59	90.36	83.24
1st Cl. Conc	0.76	8.894	25.305	74.542	33.226	96.06	88.05	82.21
1st Cl. Tail	2.15	0.050	14.218	0.690	0.143	1.53	0.03	1.03
2nd Cl. Tail	0.38	0.174	20.480	1.649	1.179	0.93	2.38	1.49
3rd Cl. Conc	0.38	17.431	30.029	145.912	62.625	95.13	87.08	80.72
Comb. Knelson Conc & Column Conc	0.50	13.372	26.736	121.768	54.782	95.87	95.46	92.75
Combined Final Tail	99.50	0.003	3.115	0.029	0.022	4.13	4.54	7.25

Zone 87 composite also performed well in the pilot plant as expected. For the Zone 87 ore sample (as-received milled feed), an overall recovery of 95.87%, 95.46%, and 92.75% for copper, gold, and silver, respectively, was achieved. The combined rougher-scavenger and cleaner circuit copper, gold, and silver flotation recoveries (based on the as-received milled Knelson feed) were approximately 95.13% at a copper grade of 17.43%, 87.08% at a gold grade of 145.9 g/t, and 80.72% at a silver grade of 62.6 g/t, respectively. The combined rougher-scavenger and cleaner KAX 51 collector, SPRI 206 collector and Na_2SO_3 depressant dosages, based on rougher feed, were approximately 51 g/t, 28 g/t and 100 g/t, respectively.

Note that the Knelson gravity recovery from this pilot testwork program was lower than predicted by the E-GRG test results. This is due to the configuration of the gravity concentrator in the pilot plant as seen in Figure 13-14 where a 20% portion of the ball mill discharge was fed to the gravity concentrator (3" Knelson). In normal plant operation, gravity concentration is achieved by treating a portion of the circulating load (cyclone underflow) not the ball mill discharge. The pilot plant configuration treating the ball mill discharge was necessary to avoid dilution of the ball mill feed caused by the excess amount of fluidization water generated by the 3"Knelson. While the pilot plant circuit configuration is sub-optimal it did provide gravity recovery prior to flotation as intended.

A brief economic analysis was conducted on the flotation tails to verify if further treatment is required. The results from the analysis did not justify the need to further treat the flotation tails due to their low grades. However, the flotation tails were still sent to Base Met for cyanidation testwork for the purpose of testwork completion.

13.7.2 Eriez Testwork for J, SW1, SW2, and 87 Zone (800 kg Samples)

A summary of the Eriez testwork results for the 800 kg samples are presented Table 13-17.

Table 13-17 Combined Rougher-Scavenger and Cleaner Flotation Results (800 kg Samples)

Zone	Cleaner Feed P_{80} (µm)	Streams	Mass Yield %	Assays							Recovery (%)		
				Cu %	Fe %	S %	Au g/t	Ag g/t	Zn %	As %	Cu	Au	Ag
J	20	Tails	99.61	0.04	3.64	0.57	0.036	0.141	0.003				
		Feed	100	0.059	3.78	0.72	0.371	0.772	0.004				
		R/S Conc	3.91	1.44	19.77	17.89	8.82	17.27	0.021				
		Conc	0.39	14.09	38.52	38.38	86.90	163.49	0.14	0.01	93.03	90.39	81.79
SW-1	20	Tails	99.66	0.005	4.37	0.31	0.046	0.138	0.010	0.011			
		Feed	100	0.052	4.48	0.45	0.388	0.762	0.012	0.014			
		R/S Conc	2.66	1.88	20.96	16.27	13.56	27.68	0.273				
		Conc	0.34	13.71	35.90	42.40	100.30	183.02	0.62	0.72	89.98	88.17	81.98
SW-1	13.5	Tails	99.70	0.004	4.37	0.33	0.039	0.118	0.008	0.012			
		Feed	100.00	0.052	4.48	0.45	0.388	0.762	0.012	0.014			
		R/S Conc	2.66	1.88	20.96	16.27	13.56	27.68	0.273				
		Conc	0.30	15.9	38.90	41.80	114.72	211.84	1.23	0.68	93	89.87	84.55
SW-2 High As	20	Tails	99.85	0.033	4.29	1.34	0.078	0.357	0.012	0.115			
		Feed	100	0.03	4.33	1.38	0.422	0.695	0.016	0.115			
		R/S Conc	4.24	0.68	32.07	30.09	9.43	14.90	0.267	2.65			
		Conc	0.15	18.16	31.37	37.50	228.62	224.75	2.58	0.13	91.08	81.46	48.65
SW-2 High As	13.5	Tails	99.81	0.02	4.29	1.34	0.06	0.219	0.013	0.116			
		Feed	100	0.03	4.33	1.38	0.422	0.695	0.016	0.115			
		R/S Conc	4.24	0.68	32.07	30.09	9.43	14.90	0.27	2.65			
		Conc	0.19	14.49	35.80	37.50	189.97	249.67	1.58	0.05	92.16	85.84	68.50
87	20	Tails	99.62	0.007	3.23	0.34	0.03	0.03	0.022	0.01			
		Feed	100	0.072	3.34	0.46	0.353	0.333	0.032	0.013			
		R/S Conc	3.00	2.27	18.92	13.62	10.89	10.18	0.87	386.04			
		Conc	0.38	17.34	32.08	31.53	86.21	79.89	2.45	0.76	90.16	91.72	90.1
87	13.5	Tails	99.61	0.006	3.23	0.33	0.028	0.033	0.024	0.009			
		Feed	100	0.072	3.34	0.46	0.353	0.333	0.032	0.013			
		R/S Conc	3.00	2.27	18.92	13.62	10.89	10.18	0.87	386.04			
		Conc	0.39	16.86	31.81	32.66	83.23	76.89	1.97	0.93	91.09	91.96	90.07

Note: R-S Feed P_{80} = 75 µm for all samples.

From Table 13-17, under the optimal flotation conditions of each zone ores, the combined circuit recoveries of 90.0-94.7%, 85.8-92.0%, and 68.5-90.1% were achieved for copper, gold, and silver, respectively.

For all the four samples (J, SW1, SW2 and 87), the optimal rougher-scavenger feed particle size distribution determined from batch flotation testing was P_{80} 75 µm. Additionally, the optimal cleaner re-grind size was P_{80} 20 µm, except for SW1 and SW2 where finer grinding at P_{80} 13.5 µm showed slightly improved the flotation performance. SW ore will be blended in with other ore types for the mill feed to lessen its impact in the flotation performance at a regrind size of P_{80} 20 µm.

For J-Zone ore sample, an overall rougher-scavenger-cleaners flotation recovery of 93.0%, 90.4%, and 81.8% was achieved for copper, gold, and silver, respectively. The final concentrate grades were 14.1%, 86.9 g/t, and 163.5 g/t for copper, gold, and silver, respectively. The overall KAX 51 collector, SPRI 206 collector and Na_2SO_3 depressant dosages were approximately 47 g/t, 27 g/t and 400 g/t, respectively.

For SW1-zone ore sample, an overall rougher-scavenger-cleaners (P_{80} 13.5 µm) flotation recovery of 93.0%, 89.9% and 84.6% was achieved for copper, gold, and silver, respectively. The final concentrate grades were 15.9%, 114.7 g/t, and 211.8 g/t for copper, gold, and silver, respectively. The overall KAX 51 collector, SPRI 206 collector and Na_2SO_3 depressant dosages were approximately 48.6 g/t, 32 g/t and 250 g/t, respectively.

For SW2-zone ore sample, an overall rougher-scavenger-cleaners (P_{80} 13.5 µm) flotation recovery of 92.2%, 85.8%, and 68.5% was achieved for copper, gold, and silver, respectively. The final concentrate grades were 14.5%, 190.0 g/t, and 249.7 g/t for copper, gold, and silver, respectively. The overall KAX 51 collector, SPRI 206 collector, Na_2SO_3 depressant and Tennaclean 163 Arsenopyrite depressant dosages were approximately 56.1 g/t, 35.1 g/t, 350 g/t, and 350 g/t, respectively.

For 87-Zone ore sample, an overall rougher-scavenger-cleaners (P_{80} 20 µm) flotation copper recovery of 90.2%, 91.7%, and 90.1% was achieved for copper, gold, and silver, respectively. The final concentrate grades were 17.3%, 86.2 g/t, and 79.9 g/t for copper, gold, and silver, respectively. The overall KAX 51 collector, SPRI 206 collector, Na_2SO_3 depressant, and $ZnSO_4$ depressant dosages were approximately 55.1 g/t, 34.1 g/t, 400 g/t, and 150 g/t, respectively.

13.7.3 Eriez Testwork Results for X22-1 and X22-2 (150 kg Samples)

Preliminary flotation testwork on X22 were limited to only one rougher-scavenger feed at P_{80} 75 µm and one cleaner feed at P_{80} 13.5 µm, due to the small sample mass available (150 kg each).

Table 13-18 summarizes the balanced benchtop mechanical cell rougher-scavenger and cleaner flotation test results for the X22-1 and X22-2 samples.

Table 13-18: Combined Rougher-Scavenger and Cleaner Flotation Results

Zone	Cleaner Feed P80 µm*	Streams	Mass Yield %	Assays							Recovery (%)		
				Cu %	Fe %	S %	Au g/t	Ag g/t	Zn %	As %	Cu	Au	Ag
X22-1 (150 kg)	13.5	Tail		0.01	2.28	0.2	0.092	0.575	0.003	0.000			
		Feed		0.09	2.45	0.39	0.557	3.428	0.006	0.000			
		Conc	0.56	14.45	31.8	34	83.4	510.99	0.49	0.013	89.31	83.61	83.31
X22-2 (150 kg)	13.5	Tail		0.004	1.85	0.4	0.094	0.23	0.008	0.001			
		Feed		0.025	1.89	0.46	0.41	0.589	0.01	0.001			
		Conc	0.16	13.51	28.82	34.47	201.81	229.22	1.79	0.032	83.33	77.21	61.05

* R-S feed is P_{80} 75 um for all samples.

The recoveries in Table 13-18 were sub-optimal as Eriez added an arsenic depressant believing arsenic level was a problem when in fact the level was very low. The arsenic depressant was not used in the 1,500 kg samples for X22 tested later.

13.7.4 Eriez Testwork Results for X22-1 and X22-2 (1,500 kg Samples)

Two representative composite feed samples of crushed assay rejects from X22-1 and X22-2 zones, each weighing ~1,500 kg were shipped to Eriez to investigate the recovery of gold, copper, and silver.

Table 13-19 summarizes the mass balanced column rougher-scavenger and benchtop cleaner flotation test results for the two samples of X22-1 and X22-2 zone ore.

Table 13-19: Combined Rougher-Scavenger and Cleaner Flotation results

Zone	Cleaner Feed P_{80} µm*	Streams	Mass Yield %	Assays							Recovery (%)		
				Cu %	Fe %	S %	Au g/t	Ag g/t	Zn %	As g/t	Cu	Au	Ag
X22-1	20 (Open Circuit)	Combined Tail	99.54	0.004	2.19	0.27	0.028	0.158	0.005	4.45			
		Feed	100.0	0.069	2.34	0.44	0.327	1.11	0.011	4.79			
		R/S Con	6.00	1.131	9.33	6.91	5.016	18.065	0.136	38.3	98.05	92.07	97.65
		Conc	0.46	14.24	34.51	37.2	65.59	209.05	1.27	80.5	93.79	91.4	85.8
X22-1	20 (Closed Circuit)	Combined Tail	99.55	0.004	2.18	0.27	0.028	0.142	0.005	4.45			
		Feed	100.0	0.069	2.34	0.44	0.327	1.11	0.011	4.79			
		R/S Con	6.00	1.131	9.33	6.91	5.016	18.065	0.136	38.3	98.05	92.07	97.65
		Conc	0.45	14.35	36.66	37.48	65.92	213.55	1.27	80.6	94.03	91.42	87.22
X22-2	20 (Open Circuit)	Combined Tail	99.86	0.002	1.83	0.24	0.029	0.092	0.007	6.54			
		Feed	100.0	0.024	1.88	0.29	0.324	0.476	0.009	6.83			
		R/S Con	1.29	1.79	24.03	20.75	23.469	34.829	0.498	241.5	95.01	93.32	94.15
		Conc	0.14	16.01	34.64	36.44	214.7	279.68	1.9	220.3	90.72	91.14	80.72
X22-2	20 (Closed Circuit)	Combined Tail	99.85	0.002	1.83	0.23	0.023	0.091	0.007	6.59			
		Feed	100.0	0.024	1.88	0.29	0.324	0.476	0.009	6.83			
		R/S Con	1.29	1.79	24.03	20.75	23.469	34.829	0.498	241.5	95.01	93.32	94.15
		Conc	0.15	15.09	34.49	36.91	205.7	263.1	1.67	174.1	91.09	93.03	80.9

* R-S feed is P_{80} 75 um for all samples.

For X22-1 sample, an overall rougher-scavenger and 3-stage cleaner (closed-circuit) column flotation recovery of 94.0%, 91.4% and 87.2% was achieved for copper, gold, and silver, respectively. The final concentrate grades were 14.35%,65.9 g/t and 213.6 g/t for copper, gold, and silver, respectively. The overall KAX 51 collector, SPRI 206 collector and Na_2SO_3 depressant dosages were approximately 55.1 g/t, 33 g/t and 100 g/t, respectively.

For X22-2 sample, an overall rougher-scavenger and 3-stage cleaner (closed-circuit) column flotation recovery of 91.1%, 93.0% and 80.9% was achieved for copper, gold, and silver, respectively. The final concentrate grades were 15.1%, 205.7 g/t and 263.1 g/t for copper, gold, and silver, respectively. The overall KAX 51 collector, SPRI 206 collector, Na_2SO_3 depressant, and $ZnSO_4$ depressant dosages were approximately 64.0 g/t, 38.0 g/t, 100 g/t, and 60 g/t, respectively.

Under optimal flotation conditions of each zone ores, the combined circuit recoveries of 91.1-94.0%, 91.4-93.0%, and 80.9-87.2% were achieved for copper, gold, and silver, respectively.

The overall pilot plant test results for all samples are presented below in Table 13-20. The pilot plant campaign succeeded at achieving optimal recoveries for gold, copper, and silver.

Table 13-20: Overall Pilot Plant Test Results

Zone	Sample	Sample Head Assay			Final Tails Assay			Overall Recovery		
		Cu, %	Au, g/t	Ag, g/t	Cu, %	Au, g/t	Ag, g/t	Cu, %	Au, %	Ag, %
J	3,000 kg	0.059	0.434	0.836	0.006	0.035	0.096	89.83	91.94	88.52
J	800 kg Composite	0.059	0.371	0.772	0.004	0.036	0.141	93.22	90.30	81.74
SW	3,000 kg	0.065	0.624	1.123	0.004	0.078	0.124	93.85	87.50	88.96
SW	800 kg Composite	0.052	0.388	0.762	0.005	0.046	0.138	90.38	88.14	81.89
87	3,000 kg	0.070	0.590	0.262	0.003	0.029	0.022	95.71	95.08	91.60
87	800 kg Composite	0.072	0.460	0.353	0.007	0.030	0.030	90.28	93.48	91.50
X22-1	1,500 kg	0.069	0.327	1.110	0.004	0.028	0.142	94.20	91.44	87.21
X22-2	1,500 kg	0.024	0.324	0.476	0.002	0.023	0.091	91.67	92.90	80.88

*Small X22 samples excluded in overall results.

13.8 Base Metallurgical Laboratories Testwork (2022 and 2024)

The metallurgical testing program at Base Met was intended to support the FS phase using samples from the Eriez pilot test program. A summary of key results from the program is presented in the subsections to follow. Comminution testwork results has been summarized in section 13.5 and will not be part of this section.

13.8.1 Chemical Content of the Feed Samples (2022)

There were several types of chemical analyses performed on the initial three composite head samples (part of 3,000 kg program for J, SW and 87). Results for the head assay are summarized in Table 13-21. Gold in the sample measured 0.49 to 0.66 g/t, while sliver measured 0.9 to 1.6 g/t. Sulphur content was low between 0.46 and 0.92%.

Table 13-21: Head Sample Chemical Content

Sample ID	Assays			
	Au, g/t	Ag, g/t	Fe, %	S, %
SW Domain	0.66	1.6	6.0	0.92
Zone 87	0.49	1.1	2.7	0.46
Zone J4/J5	0.66	0.9	4.3	0.90

13.8.2 Mineral Content of the Feed Samples (2022)

The mineral content of the feed samples was measured by QEMSCAN. The samples were assessed using Particle Mineral Analysis (PMA) protocols on ground samples screened into four fraction sizes. A summary of the mineral content along with copper mineral distribution is shown in Table 13-22 and Table 13-23.

The samples consisted mainly of silicates, including feldspars, quartz, and biotite/phlogopite. Sulphide minerals accounted for 1.1% to 2.3% of the total sample mass, mainly iron sulphides. Copper sulphides, dominated by chalcopyrite, accounted for 0.2% to 0.3% of the samples mass. Secondary copper minerals were measured in all composites at lower concentrations. Arsenopyrite was also detected in the SW Zone. The iron sulphide to copper ratio ranged between 3.8 to 9.0. Implementing conditions to suppress pyrite would be prudent to producing a clean concentrate.

Table 13-22: Mineral Content (Head Samples)

Class	Mineral	Mineral Content (%)		
		SW Domain	Zone 87	Zone J4/J5
Sulphides	Chalcopyrite	0.32	0.22	0.18
	Bornite	0.02	0	<0.01
	Chalcocite / Covellite	0.01	<0.01	<0.01
	Tetrahedrite / Tennantite	<0.01	0	<0.01
	Cuprite	<0.01	<0.01	<0.01
	Galena	0.01	0.02	0.05
	Sphalerite	0.04	0.04	0.05
	Pyrite	0.83	0.57	1.15
	Pyrrhotite	1.03	0.28	0.50
	Arsenopyrite	0.05	<0.01	<0.01
	Other Sulphides	<0.01	<0.01	<0.01
Silicates, Oxides, Carbonates and Others	Iron Oxides	3.41	0.44	0.65
	Ilmenite	1.65	0.05	0.08
	Quartz	21.0	26.0	20.1
	Plagioclase Feldspar	42.0	39.9	43.4
	K-Feldspar	2.43	3.56	2.49
	Biotite / Phlogopite	11.25	10.7	16.7
	Amphibole (Actinolite)	2.09	6.31	7.15
	Chlorite	2.07	2.18	2.95
	Muscovite	6.76	5.91	1.29
	Epidote	1.36	1.96	1.81
	Pyroxene (Augite)	0.03	0.08	0.03
	Andradite	0.29	0.22	0.20
	Kaolinite (Clay)	0.08	0.05	0.03
	Calcite	1.74	0.73	0.29
	Dolomite / Ankerite	0.10	0.01	0.01
	Ti Minerals	1.08	0.47	0.41
	Apatite	0.29	0.22	0.34
	Others	0.11	0.07	0.07
Total		**100**	**100**	**100**

Table 13-23 Copper Sulphide Distribution (Head Samples)

Metal	Mineral	Distribution (%)		
		SW Domain	Zone 87	Zone J4/J5
Copper	Chalcopyrite	85.1	96.5	97.5
	Bornite	9.3	0	<0.1
	Chalcocite / Covellite	5.2	3.3	2.2
	Tetrahedrite / Tennantite	0.2	0	<0.1
	Cuprite	0.2	0.2	0.2
	Total	**100**	**100**	**100**

13.8.3 Mineral Liberation for Feed Samples (2022)

The QEMSCAN PMA was performed on the feed samples ground to 67 to 80 µm K_{80}, to provide a measurement of mineral liberation. For these samples, four size fractions were analyzed. The resulting data is displayed in Table 13-24.

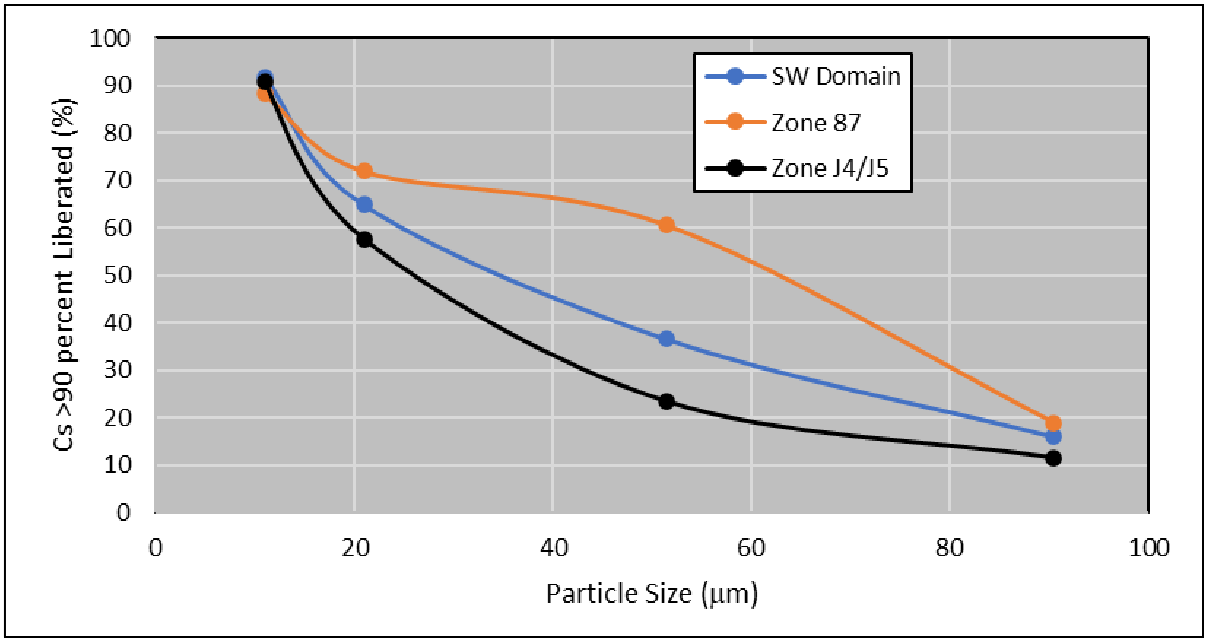
Table 13-24: Mineral Liberation Data

Mineral Status	Mineral Liberation 2D - Percent														
	SW Domain – 67µm					Zone 87 – 81µm					Zone J4 / J5 - 80µm				
	Cp	Ga	Sp	Py	Gn	Cp	Ga	Sp	Py	Gn	Cp	Ga	Sp	Py	Gn
Liberated	59	56	58	77	100	55	52	56	69	100	53	53	50	72	100
Binary – Cp		0	4	2	<1		4	2	1	<1		1	1	1	<1
Binary – Ga	0		2	<1	<1	1		2	<1	<1	<1		10	1	<1
Binary – Sp	<1	4		1	<1	1	5		<1	<1	<1	7		1	<1
Binary – Py	2	9	7		<1	2	12	2		<1	4	17	8		<1
Binary – Gn	36	12	20	19		37	16	16	27		40	6	15	23	
Multiphase	2	19	9	2	<1	3	11	23	3	<1	2	16	16	2	<1
Total	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**

Copper sulphides in the three samples was between 53% and 59% liberated at between 67 µm and 80 µm K_{80}. Based on measurements of operating mines, this level of liberation is sufficient for adequate recovery to a rougher concentrate. Most of the unliberated copper sulphides were locked in binary form with non-sulphide gangue minerals.

Figure 13-17 displays the mineral release curves for copper sulphides. Typically, regrind target size would be achieved when 85% to 95% of the target minerals at a specific size have a mineral liberation value of greater than 90%. For these samples, ideal liberation values are not achieved until the particle sizes are below approximately 12 µm.

Figure 13-17: Copper Sulphide Mineral Release



Source: Base Met (2022)

13.8.4 Mineralogical Analysis for Rougher Scavenger Concentrates (2022)

The mineral content and liberation of the rougher scavenger concentrate samples was measured by QEMSCAN via a PMA routine, on samples screened into four fraction sizes. A summary of the results is displayed in Table 13-25 and Table 13-26 and Figure 13-18 to Figure 13-20.

Copper sulphides in the concentrates, dominated by chalcopyrite, accounted for 3.3% to 7.0% of the samples. Pyrite and pyrrhotite content in these samples were very high, particularly in the J Zone R/S Concentrate. Iron sulphides in the SW, 87 and J zone R/S concentrates measured 20%, 31% and 54%, respectively. This corresponds to iron sulphide to copper sulphide ratios of 16, 23 and 48. At these values, producing a clean copper concentrate may prove challenging. Testing and optimization of conditions may assist to better reject pyrite and pyrrhotite from the rougher circuit.

Copper sulphide liberation in these concentrates ranged between 65% to 71 %. As mentioned in the feed mineralogy section, copper sulphide liberations of about 90 percent are required to produce a high-grade copper concentrate. Regrinding of the rougher concentrate will be required.

Pyrite minerals in these concentrates were between 86% and 91% liberated and interlocking with copper sulphide minerals were low at 2% to 3% of the total pyrite. The majority of the liberated pyrite particles were coarser than 33 µm K_{80} and with adequate conditions, should be rejected from the rougher concentrates.

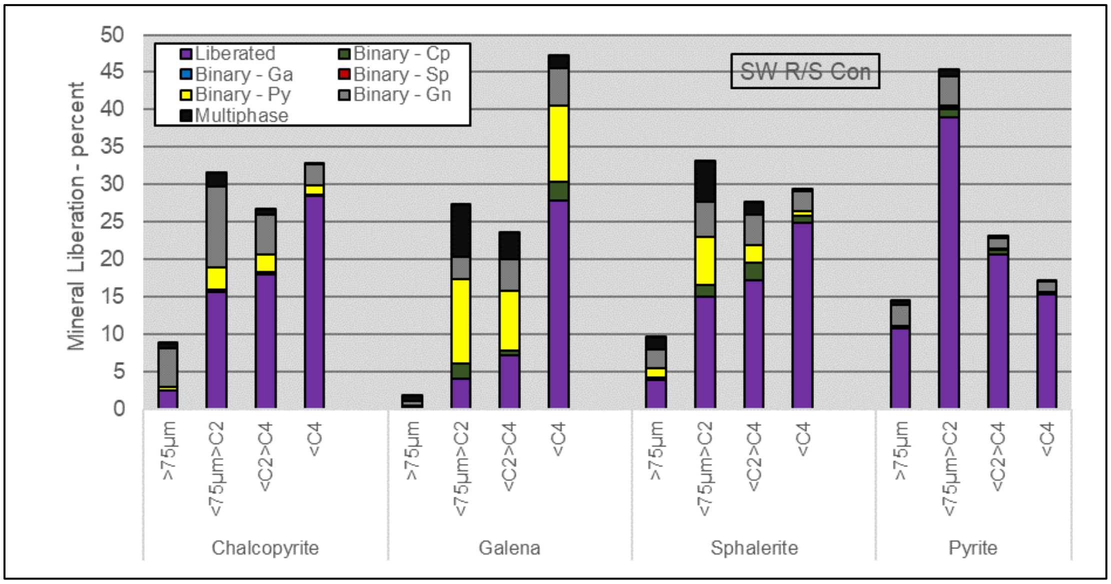
Table 13-25: Mineral Content (R/S Concentrate)

Class	Mineral	Mineral Content - Percent		
		SW R/S Concentrate	Zone 87 R/S Concentrate	J Zone R/S Concentrate
Sulphides	Chalcopyrite	3.15	6.98	5.90
	Bornite	0.14	<0.01	0.00
	Chalcocite / Covellite	0.05	0.04	0.04
	Tetrahedrite / Tennantite	<0.01	<0.01	0.00
	Cuprite	<0.01	0.00	0.00
	Galena	0.01	0.02	0.02
	Sphalerite	0.24	1.25	0.06
	Pyrite	10.2	11.13	42.5
	Pyrrhotite	5.59	11.8	5.27
	Arsenopyrite	0.50	0.04	0.00
	Other Sulphides	0.03	0.04	0.05
Silicates, Oxides, Carbonates and Others	Iron Oxides	1.42	1.03	1.09
	Ilmenite	0.99	0.14	0.06
	Quartz	18.4	16.1	9.4
	Plagioclase Feldspar	31.0	28.3	18.2
	K-Feldspar	2.19	3.80	2.02
	Biotite / Phlogopite	9.78	4.8	5.6
	Amphibole (Actinolite)	4.18	3.90	3.02
	Chlorite	2.99	2.53	2.96
	Muscovite	4.76	4.49	0.89
	Epidote	1.61	1.00	1.35
	Pyroxene (Augite)	0.01	0.00	0.07
	Andradite	0.26	0.34	0.21
	Kaolinite (Clay)	0.08	0.08	0.05
	Calcite	1.22	1.35	0.50
	Dolomite / Ankerite	0.03	0.05	0.00
	Ti Minerals	0.71	0.36	0.36
	Apatite	0.25	0.19	0.13
	Others	0.13	0.22	0.15
	Total	**100**	**100**	**100**

Table 13-26: Mineral Liberation (R/S Concentrate)

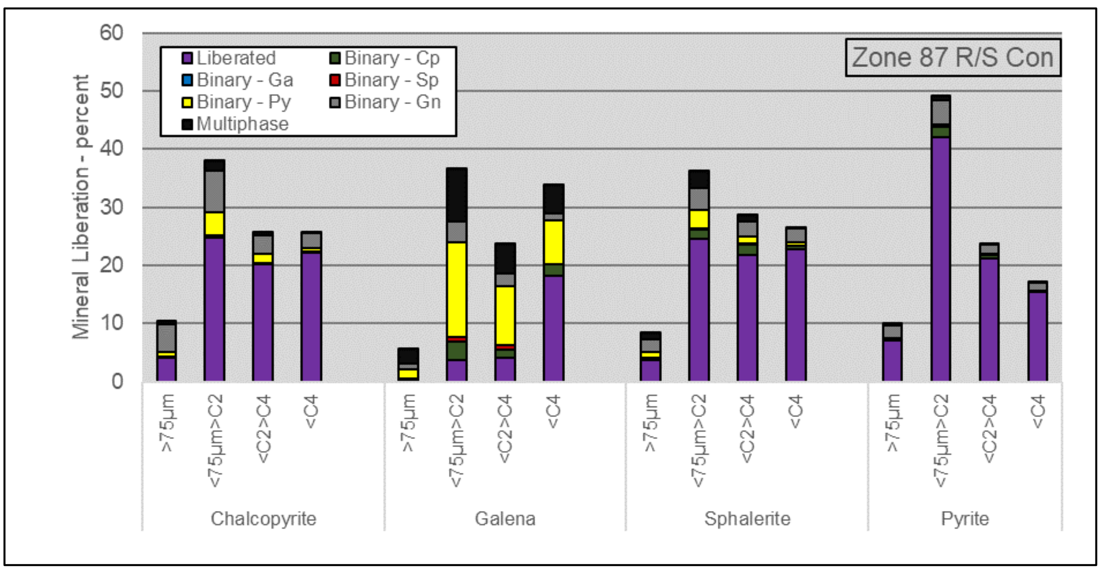
Mineral Status	Mineral Liberation 2D - Percent														
	SW R/S Con – 51µm					Zone 87 R/S Con – 44µm					J Zone R/S Con - 60µm				
	Cp	Ga	Sp	Py	Gn	Cp	Ga	Sp	Py	Gn	Cp	Ga	Sp	Py	Gn
Liberated	65	39	61	86	97	71	26	73	86	95	65	37	50	91	89
Binary – Cp		5	5	2	1		7	4	3	2		5	10	2	3
Binary – Ga	<1		0	<1	<1	<1		<1	1	<1	<1		1	<1	<1
Binary – Sp	<1	0		<1	2	1	2		<1	3	<1	<1		<1	7
Binary – Py	7	30	11		<1	7	36	6		<1	10	32	13		<1
Binary – Gn	24	13	14	10		18	8	11	9		21	9	12	6	
Multiphase	3	13	9	2	<1	3	22	6	2	<1	3	17	14	1	1
Total	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**

Figure 13-18: SW-Zone Mineral Liberation by Size and Class (R/S Concentrate)



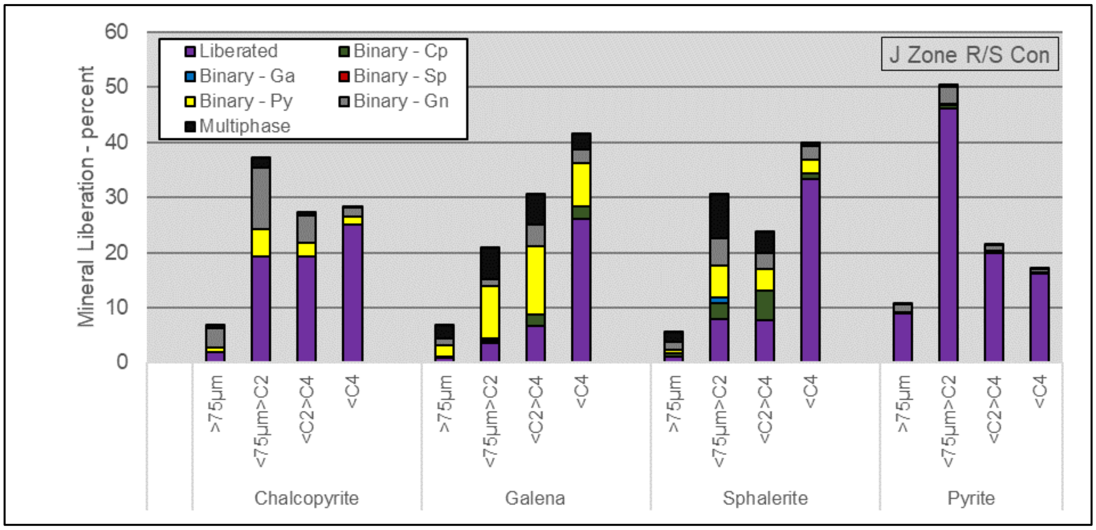
Source: Base Met (2022)

Figure 13-19: Zone 87 Mineral Liberation by Size and Class (R/S Concentrate)



Source: Base Met (2022)

Figure 13-20: J Zone Mineral Liberation by Size and Class (R/S Concentrate)



Source: Base Met (2022)

13.8.5 Cyanide Leach Test Results – Cleaner Tailings

Three additional samples were provided for cyanide leach tests. These were comprised of J-Zone cleaner tails, SW-Zone final cleaner tails, and zone 87 combined tails. Two cyanide leach tests were performed on the J-Zone cleaner tails, evaluating oxygen versus air sparging. The remaining tests were performed with oxygen. The results are summarized in Table 13-27.

Tests were performed for a duration of 24 h, with sample points at 2, 6, 8 and 24 h. Tests were conducted at pH 10.5, using 5,000 ppm NaCN.

Table 13-27: Cyanide Leach Test Results

Composite	Test	O_2/Air Sparged	Au Leach Extraction - % Cum. Time (Hours)				24-hr Extraction %		CNTL g/t	Consumption kg/t	
			2	6	8	24	Ag	Cu	Au	NaCN	Lime
J-Zone Cleaner Tails	2	Air	91.5	90.7	92.3	91.4	33.5	12.0	0.21	10.1	4.05
J-Zone Cleaner Tails	3	O_2	92.7	90.3	93.5	93.4	50.6	11.7	0.16	9.70	4.10
SW Zone Final Cleaner Tails	6	O_2	83.3	92.0	88.1	84.0	57.2	58.6	0.05	4.80	2.05
87 Zone Combined Tails	7	O_2				94.8	64.1	65.6	0.06	21.7	2.05

Initially, two tests performed on the J-Zone cleaner tails, evaluating air versus oxygen sparging, resulted in higher overall extraction of both gold and silver and lower NaCN consumption, using oxygen. With oxygen sparging, gold from the tailing samples were between 84% and 95% extracted to solution. Silver from the tailing samples was between 51% and 64% extracted to solution. Cyanide consumption was high for these samples, ranging between 4.8 kg/t and 21.7 kg/t. Lime consumption ranged between 4.1 kg/t and 2.1 kg/t. Secondary copper minerals may partially contribute to the high cyanide contents.

13.8.6 Base Met Dewatering Tests (2022)

A series of dewatering tests were performed on tailing and concentrate samples provided by Eriez. Results are presented in Table 13-28 to Table 13-30.

Rougher tailings and cleaner tailings were received for each of the three composites. The Rougher and cleaner tailings were combined at ratios provided by Eriez to represent final tailings. Note that cleaner tails were not subjected to cyanidation for this testwork.

Initially, flocculant scoping tests were performed on J zone tailings evaluating five various anionic, non-ionic, and cationic flocculants at a dosage of 20 g/t. Final density ranged between 18% and 58% solids. Settling rates ranged between 0.3 and 4.4 mm/s. Magnafloc 10 was chosen as the preferred flocculant and the remaining dewatering tests were performed using MF10.

Static settling tests followed. These were conducted in 1 L cylinders, using MF10 as the flocculant, to assess flocculant dosage at various pH levels. Overall, higher pH resulted in improved settling rates, but lower final density. The static tests were followed by dynamic settling tests, investigating various loading rates at varying flocculant dosages.

The sheared viscosity parameters of the tailing samples were determined using a Brookfield DV2T viscometer. Slurry viscosity of less than 100 cps, at a shear rate of 120.5 sec-1 is considered acceptable for pumping applications. Yield stress was measured between 18 and 89 Pa.

Table 13-28: Flocculant Scoping Tests

Sample ID	Test	pH	Initial Density % Solids	Floc	Floc Type	Dosage g/t	Final %Solids	Settling Rate mm/s
Combined Tails	F1	8.4	15	MF10	Anionic	20	54.7	3.79
		8.4	15	MF380	Cationic	20	17.6	0.30
		8.4	15	MF351	Non-Ionic	20	58.4	2.11
		8.4	15	MF156	Anionic	20	54.7	4.08
		8.4	15	AN913SH	Anionic	20	51.2	4.38

Table 13-29: Static Settling Tests

Composite	Test	Flocculent		pH	Settling Rate mm/s	Final Density % Solids
J-Zone R/S Tail +Cleaner Tails	S1	MF10	20	9.0	2.78	66.6
	S2	MF10	20	10.0	3.25	64.6
	S3	MF10	20	11.0	3.45	61.2
	S4	MF10	30	11.0	4.03	57.7
	S5	MF10	40	11.0	5.64	53.8
	S6	MF10	40	8.4	6.38	66.6
	S7	MF10	40	10.0	9.23	63.1
SW Scavenger / Cleaner Tail Blend	S8	MF10	30	9.0	5.9	67.2
	S9	MF10	30	10.0	5.5	66.4
	S10	MF10	30	11.0	5.7	67.0
	S11	MF10	20	11.0	4.3	64.3
	S12	MF10	40	11.0	5.2	63.6
87 Zone Cleaner / Scavenger Tail Blend	S13	MF10	30	9.0	5.4	65.1
	S14	MF10	30	10.0	8.2	62.1
	S15	MF10	30	11.0	8.3	60.1
	S16	MF10	20	11.0	7.8	62.7
	S17	MF10	40	11.0	7.6	60.7

Table 13-30: Dynamic Settling Tests

Test	Composite	Test	Loading Rate $t/m^2/hr$	Floc Dosage g/t	U/F Density % Solids	Unsheared Yield Stress Pa	pH	Turbidity FAU
D1	J-Zone R/S Tail +Cleaner Tails	D1-A	0.50	30	57.2	81	11	58
		D1-B	0.70	30	59.4	63	11	63
		D1-C	1.00	30	56.7	44	11	48
		D1-D	1.00	40	54.1	30	11	52
		D1-E	1.00	20	58.7	46	11	123
		D1-F	1.20	30	53.9	32	11	73
D2	SW Scavenger / Cleaner Tail Blend	D2-A	0.30	30	61.3	62	11	2
		D2-B	0.50	30	61.1	61	11	15
		D2-C	0.70	30	60.5	72	11	22
		D2-D	0.70	20	61.4	61	11	25
		D2-E	0.70	40	56.9	89	11	38
D3	87 Zone Cleaner / Scavenger Tail Blend	D3-A	0.70	30	60.2	18	9	342
		D3-B	0.50	30	65.3	45	11	44
		D3-C	0.70	30	59.8	40	11	47
		D3-D	1.00	30	57.5	45	11	16
		D3-E	0.70	20	61.7	35	11	6

13.8.7 Pressure Filtration – Concentrates

Scoping level pressure filtering tests were conducted in a 45 mm diameter cylinder concentrate samples for J, SW and 87. Blow times of 30, 60, and in some cases, 85 seconds were assessed. Moisture contents of 9-10% were achieved, but at the longest blow time and thinnest filter cake, resulting in low filter rates. About 15% moisture was achieved at quicker filtration rates. Due to the scoping level assessment and small mass provided, additional testing and optimization may be necessary. Results are presented in Table 13-31.

Table 13-31: Pressure Filtration Tests for J, SW and 87 Concentrates

Sample ID	Test	pH	Sample Mass	Feed Pressure	Blow Time – Sec		Cake Thickness	Cake Moisture	Filter Rate
			Grams	Psi	Total	Filter Time	Mm	%	$Kg/m^2/hr$
J-Zone 3rd Cleaner Conc.	PF-1	7.8	30	80	60	10	9	17.1	822
			30	80	180	11	9	13.4	284
			30	80	300	10	8	9.2	187
			60	80	60	28	16	16.3	1939
			60	80	180	28	18	13.3	674
			60	80	300	25	19	12.4	395
			85	80	180	49	27	15.0	943
Southwest Cleaner Conc.	PF-2	7.6	30	80	60	2	8	12.4	796
			30	80	180	2	8	13.9	250
			30	80	300	2	7	10.0	163
			60	80	60	5	15	15.3	1841
			60	80	180	6	15	10.0	590
			60	80	300	6	15	11.5	361
87 Zone Cleaner Conc.	PF-3	5.9	30	80	60	2	8	11.4	819
			30	80	180	2	8	9.7	270
			30	80	300	2	8	9.1	165
			60	80	300	15	19	10.1	384
			85	80	300	21	25	11.6	542
			60	80	60	9	18	14.1	1913
			60	80	180	9	19	10.3	660

13.8.8 Minor Elements for Concentrates

The minor element content of the concentrates was determined via several different digestions and assay techniques. For all three composites, this analysis was conducted on the copper concentrate provided. A summary of the results is reported in Table 13-32. A review of the data revealed the following salient points:

The SW Zone Cleaner Con, Z87 Zone Cleaner Con and J-Zone 3rd Cleaner Con measured 104, 131 and 111 g/t gold, respectively. Silver was measured at 309, 140 and 340 g/t, respectively, in these concentrates. Copper ranged from 14.9% to 18.1% and is generally below typically saleable copper concentrate grades. However, the gold content may reduce the need for high grade concentrates to maintain quality.

Platinum and Palladium was measured low in the concentrates at between 0.03 g/t and 0.07 g/t.

Mercury measured between 0.2 g/t and 13.7 g/t. Arsenic levels measured 86 to 4,200 g/t and may be approaching smelter penalties in some cases. Consultation with a concentrate marketing specialist to advise on current penalty and payment terms for minor elements is recommended.

Table 13-32: Minor Elements (Concentrates)

Element	Units	Method	J-Zone 3rd Cleaner Con	SW Zone Cleaner Con	87 Zone Cleaner Con
Ag	g/t	AR-AA	303	309	140
As	ppm	FUS-MS-Na2O2	86	4200	544
Au	g/t	FA-AA	111	104	131
Ba	ppm	FUS-MS-Na2O2	31	37	35
Bi	ppm	FUS-MS-Na2O2	96	25	34
Ca	%	FUS-Na2O2	2.0	0.5	0.3
Cd	ppm	FUS-MS-Na2O2	64	138	158
Co	ppm	FUS-MS-Na2O2	462	1280	608
Cr	ppm	FUS-MS-Na2O2	220	190	210
Cu	%	AR-AA	14.9	18.1	16.3
F	%	FUS-ISE	0.01	<0.01	<0.01
Fe	%	AR-AA	31.5	33..8	33.5
Hg	ppm	Cold Vapour	8.7	13.7	0.2
Mg	%	FUS-Na2O2	1.38	0.14	0.1
Mn	ppm	FUS-MS-Na2O2	89	103	143
Mo	ppm	FUS-MS-Na2O2	1680	1380	775
Ni	ppm	FUS-MS-Na2O2	600	720	710
Pb	ppm	FUS-MS-Na2O2	719	1560	1560
Pd	g/t	FA-ICP	0.07	0.03	0.07
Pt	g/t	FA-ICP	0.07	0.04	0.04
S	%	LECO	38.3	39.8	39.9
Sb	ppm	AR-ICP	13	104	14
Se	ppm	FUS-MS-Na2O2	45	67	88
Si	%	FUS-Na2O2	1.49	1.00	1.28
Sr	ppm	FUS-MS-Na2O2	103	30	25
U	ppm	FUS-MS-Na2O2	1.4	1.3	1
Zn	ppm	FUS-MS-Na2O2	9120	>10000	>10000
Zn	%	AR-AA		1.37	3.1

13.8.9 Base Met Dewatering Tests (2024)

A series of dewatering tests were performed on X22-1 & XX-2 tailing samples provided by Eriez. Results are presented in Table 13-33 to Table 13-35.

Table 13-33: Flocculant Scoping Tests

Sample ID	Test	pH	Initial Density % Solids	Floc	Floc Type	Dosage g/t	Final %Solids	Settling Rate mm/s
X22-1 Final Talis	F1	-	15	MF10	Anionic	10	55.9	3.89
		-	15	MF380	Cationic	10	60.4	0.66
		-	15	MF351	Non-Ionic	10	56.7	3.66
		-	15	MF156	Anionic	10	56.7	3.49
		-	15	AN913	Anionic	10	62.4	3.64

Initially, flocculant scoping tests were performed on X22-1 zone tailings evaluating five various anionic, non-ionic, and cationic flocculants at a dosage of 10 g/t. Final density ranged between 55.9% and 62.4%

solids. Settling rates ranged between 0.66 and 3.89 mm/s. Magnafloc 10 was chosen as the preferred flocculant and the remaining dewatering tests were performed using MF10.

Table 13-34: Static Settling Tests

Composite	Test	Flocculent		pH	Settling Rate mm/s	Final Density % Solids
X22-1 Final Tails	S1	MF10	20	8.0	4.1	66.8
	S2	MF10	40	8.0	6.2	65.5
	S3	MF10	30	8.0	4.8	66.9
	S4	MF10	30	9.0	6.4	60.2
	S5	MF10	30	10.0	11.5	60.1
	S6	MF10	30	11.0	7.2	55.5

Static settling tests were conducted in 1 L cylinders, using MF10 as the flocculant, to assess flocculant dosage at various pH levels. Overall, higher pH resulted in improved settling rates, but lower final density.

Table 13-35: Dynamic Settling Test Results

Test	Composite	Test	Loading Rate $t/m^2/hr$	Floc Dosage g/t	U/F Density % Solids	Unsheared Yield Stress Pa	pH	Turbidity FAU
D1	X22-1 Final Tails	D1-A	0.50	30	62.5	26	8	951
		D1-B	0.50	40	60.8	31	8	760
		D1-C	0.70	40	56.2	95	8	403
		D1-D	1.00	40	59.5	25	8	505
		D1-E	0.50	40	60.4	34	8	784
		D1-F	0.70	60	59.9	38	8	364
D2	X22-1 Final Tails	D2-A	0.50	40	60.5	40	10	300
		D2-B	0.50	60	59.5	44	10	430
		D2-C	0.50	40	60.6	48	10	42
		D2-D	1.00	40	58.3	38	10	45
		-	-	-	-	-	-	-
D3	X22-2 Final Tails	D3-A	0.50	40	62.2	35	11	29
		D3-B	0.70	40	61.4	38	11	32
		D3-C	1.00	40	60.3	38	11	54
		D3-D	0.70	60	61.4	51	11	37
		D3-E	0.70	20	62.4	47	11	30

The dynamic settling tests investigated various loading rates at varying flocculant dosages. The results for X22 samples returned similar results to the 2022 dewatering testwork for J, SW and 87 zones.

The sheared viscosity parameters of the tailing samples were determined using a Brookfield DV2T viscometer. Slurry viscosity of less than 100 cps, at a shear rate of 120.5 sec^{-1} is considered acceptable for pumping applications. Yield stress was measured between 25 and 95 Pa.

13.9 Pocock Solid-Liquid Separation Testwork (2020)

KCA sent a sample of leached tails to Pocock for settling and viscosity testing. The as received sample, a composite of equal weights of J, Z87 and SW zones, had a P_{80} of 58 µm and a slurry pH of 8.6.

A medium to high molecular weight 5% charge density anionic polyacrylamide flocculant was selected following scouting tests. Table 13-36 presents result from the settling tests performed for combinations of feed densities and flocculant additions.

Table 13-36: Beaker Settling Test Results (Pocock, 2020)

Feed % solids	Flocc. g/t	Rise Rate $m^3 / m^2 / h$	U/flow %w/w solids	Unit Area m^2 / tpd
15	10	7.81	63	0.183
20	10	4.21	63	0.192
25	5	2.36	63	0.204
25	10	3.64	63	0.179
25	15	2.71	63	0.203

Note that the unit area includes a 25% scale-up factor, and the rise rate includes a 0.5 factor for thickener sizing. Pocock recommends using a unit area of 0.196 m^2/tpd for design purposes.

Table 13-37 present results from the settling tests performed for combinations of feed densities and shear rates.

Table 13-37: Rheology Results (Pocock, 2020)

Solids %w/w	Coefficient of Rigidity Pa.s	Yield Value Pa	Apparent Viscosity Pa. s @ Shear Rates							
			5 / sec	25 / sec	50 / sec	100 / sec	200 / sec	400 / sec	600 / sec	1,000 / sec
65.7	0.279	35.9	5.83	2.85	2.10	1.54	1.13	0.83	0.69	0.55
64.5	0.178	21.8	3.43	1.76	1.33	1.00	0.75	0.56	0.48	0.39
63.2	0.116	14.6	2.33	1.78	0.88	0.65	0.49	0.36	0.31	0.25
60.6	0.048	7.0	1.16	0.51	0.36	0.26	0.18	0.13	0.10	0.08

A decreasing apparent viscosity with an increase in shear rate (also known as 'shear thinning') is typical of pseudo plastic non-Newtonian fluids. Pocock advises to maintain the slurry density below 65% (<30 Pa yield stress) for centrifugal pumping applications.

13.10 Acid Base Accounting (ABA) Testwork

Latest tailings characterization testwork was completed by Troilus in 2023. Table 13-38 indicate that the samples tested were deemed non-PAG (not potentially acid generating). Confirmation of tailings as non-PAG eliminates the capex on acid rock drainage (ARD) containment measures.

Table 13-38: Summary of Acid Base Accounting Testwork Results

Parameter		Sample	J-Zone 1	J-Zone 2	J-Zone 3	J-Zone 4	J-Zone 5	SW-Zone 1	SW-Zone 2	SW-Zone 3	SW-Zone 4	SW-Zone 5	87-Zone 1	87-Zone 2	87-Zone 3	87-Zone 4	87-Zone 5
		Zone	J4	J4	J4	J4	J4	SW	SW	SW	SW	SW	Z87	Z87	Z87	Z87	Z87
		Lithology	Residue	Residue	Residue	Residue	Residue	Residue	Residue	Residue	Residue	Residue	Residue	Residue	Residue	Residue	Residue
		Level %S	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Carbon total	C_{total}	%	0.06	0.07	0.05	0.05	0.06	0.20	0.17	0.17	0.16	0.16	0.10	0.09	0.10	0.10	0.10
Sulfur total	S_{total}	%	0.11	0.13	0.07	0.07	0.12	0.09	0.09	0.07	0.07	0.08	0.03	0.02	0.03	0.02	0.03
Sulfates (S-GRA06)	$S_{sulfates}$	%	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Sulfates (S-GRA06a)	$S_{sulfates}$	%	<0.01	0.02	0.01	<0.01	<0.01	0.01	0.01	0.01	0.01	0.01	<0.01	0.01	0.01	0.01	0.01
Sulfides (S-IR07)	$S_{sulfides}$	%	0.10	0.10	0.06	0.07	0.12	0.08	0.09	0.07	0.07	0.07	0.03	0.03	0.03	0.02	0.02
Sulfides (Calcul)	$S_{sulfides}$	%	0.11	0.11	0.06	0.07	0.12	0.08	0.08	0.06	0.06	0.07	0.03	0.01	0.02	0.01	0.02
pH in paste	-	-	7.8	7.8	7.9	8	7.9	8.3	8.4	8.5	8.5	8.4	8.8	8.6	8.5	8.5	8.9
Neutralisation Potential	PN	kg $CaCO_3$/t	14	13	14	14	14	23	22	21	21	21	17	16	16	16	20
Maximum Acidity Potential	MPA	kg $CaCO_3$/t	3.4	4.1	2.2	2.2	3.8	2.8	2.8	2.2	2.2	2.5	0.9	0.6	0.9	0.6	0.9
Net Neutralisation Potential	PNN	kg $CaCO_3$/t	11	9	12	12	10	20	19	19	19	19	16	15	15	15	19
Ratio PN/PA	RPN	-	4.1	3.2	6.4	6.4	3.7	8.2	7.8	9.6	9.6	8.4	18.1	25.6	17.1	25.6	21.3
Potentially Acid-Generating (according to 2009 criteria)	-	-	**NPGA**	**NPGA**	**NPGA**	**NPGA**	**NPGA**	**NPGA**	**NPGA**	**NPGA**	**NPGA**	**NPGA**	**NPGA**	**NPGA**	**NPGA**	**NPGA**	**NPGA**

13.11 Results Interpretation

The following describes the main results that contributed to the development of the process design criteria for the Project:

- Comminution testwork results on SMC, abrasion, BW_i, and PILOTWAL tests conducted through JKTech, Base Met, KCA, FLSmidth, and Hazen Research Inc. were used by OMC to design the primary and secondary crushing, and for the HPGR and milling circuit. The design abrasion index value was based on an average of all available past results.
- Gravity recoveries (future) for the different ore zones were based on FLSmidth gravity recovery modelling and also on historical Troilus operation data.
- The optimal grind sizes for primary milling and regrind circuit were selected based on bench-scale flotation testwork at varying P80 by Eriez. The rougher scavenger grind is P80 75 µm and the concentrate regrind P80 is 20 µm.
- Flotation reagent consumption has been based on the latest pilot plant testwork for the individual ore zone.
- The recovery model for Troilus has been determined based on a constant tails grade from the pilot plant testwork conducted by Eriez. The recovery equations were determined to be as follows:
- Flotation Recovery % = (HG – TG) / HG x 100.

Where:

- HG = Head grade in g/t or % (value per mine plan)
- TG = Tails grade in g/t or % (value per pilot plant testwork for individual ore zone)
- base metal lab test results were used as reference for sizing concentrate and tailings dewatering equipment
- pocock solid liquid separation testwork results were used for tailings pipeline sizing
- water quality data received from Troilus was used for the design criteria

13.12 Conclusions and Recommendations

13.12.1 Conclusions

The following conclusions can be drawn from the metallurgical testwork completed on the recent samples for J, SW, 87 and X22 zones:

- Hardness testwork results classified Troilus ore to be competent with A x b value of 26.0 at the 15^{th} percent and 29.8 at the 50th percentile.
- Bond abrasion index measuring from 0.2 to 0.4 indicates that the ore is moderately abrasive.
- Crushing work index has been derived from A x b data to be 22.5 kWh/t.
- Bond ball mill work index of 13.8 kWh/t at the 85^{th} percentile and 12.1 kWh/t at the 50th percentile.

- Locked cycle PILOTWAL HPGR testwork resulted in an average m·dot value of 270 t·s/m^3·h at a net pressing force of 3.33 N/mm^2.
- All samples from the different zones had good GRG values with coarse GRG as predicted by FLSmidth gravity recovery modelling for J, SW, and. X22 zones.
- J, SW, 87 and X22 sampels all performed well in the flotation testwork conducted by Eriez.
- The optimum grind size selected was P80 75µm for the primary mill and P80 20µm for the regrind mill.
- Further treatment (leaching) of the flotation tails is not required or justifiable economically due to low flotation tails grades. No cyanide will be used anywhere in the process.
- J-Zone is expected to have an overall plant recovery of 91.6% Cu, 91.9% Au and 86.6% Ag, based on LOM head grade of 0.49 g Au/t, 1.00 g Ag/t and 0.06% Cu, respectively.
- SW-Zone is expected to have an overall plant recovery of 89.9% Cu, 87.4% Au and 82.7% Ag, based on LOM head grade of 0.49 g Au/t, 1.00 g Ag/t and 0.06% Cu, respectively.
- Zone 87 is expected to have an overall plant recovery of 91.8% Cu, 94.7% Au and 97.6% Ag, based on LOM head grade of 0.49 g Au/t, 1.00 g Ag/t and 0.06% Cu, respectively.
- Zone X22 is expected to have an overall plant recovery of 94.5% Cu, 93.1% Au and 89.9% Ag, based on LOM head grade of 0.49 g Au/t, 1.00 g Ag/t and 0.06% Cu, respectively.
- Flotation reagent consumptions for all zones combined are approximately 56 g/t KAX, 32 g/t SPRI 206, 29 g/t frother and between 100 to 400 g/t Na_2SO_3 depressant.
- Deleterious element is not expected to be present in the flotation concentrate, however, provision for zinc and arsenic depressant has been included in the design to treat ore with higher zinc or arsenic content if required.

13.12.2 Recommendations

Additional testwork is recommended to enhance confidence in the selected flowsheet and process design derived to date. This should include:

- settling and rheology testing on flotation concentrates
- regrind circuit design to be further confirmed via regrind milling testwork (i.e., jar mill test)

Note that the rougher concentrate mass pull is only 2 to 6%, resulting in very little sample mass for submitting to a laboratory for dewatering or regrind mill testwork on the concentrate. Troilus is fully aware of such limitations and has included this in the project's risk register.

Other recommendations include consultation with a concentrate marketing specialist to advise on current penalty and payment terms for minor elements.

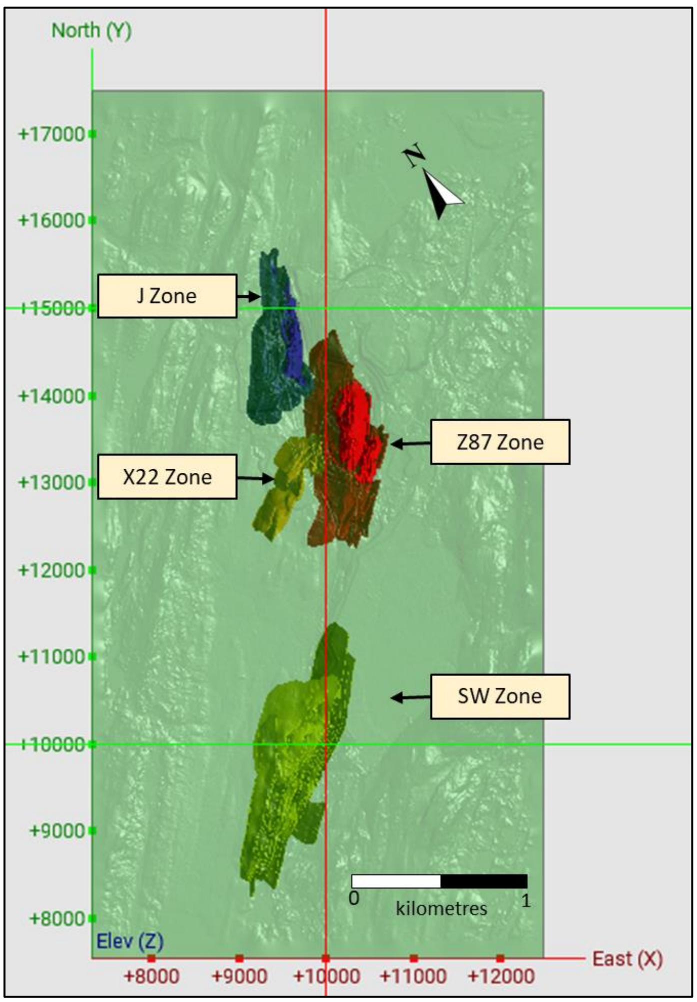
14 MINERAL RESOURCE ESTIMATES

14.1 Summary

This section discloses the mineral resources estimates for the Project, prepared, and disclosed in accordance with the CIM Standards and Definitions for Mineral Resources and Mineral Reserves (2014). The QP responsible for these resource estimates is Mr. Paul Daigle, P.Geo., Principal Resource Geologist for AGP. The effective date for these mineral resource estimates is 2 October 2023. The current mineral resources estimates for the Project include the Z87, J, X22 and SW Zones. The mineral resource estimates have been prepared using interpreted mineralized domains in each of the four deposits that comprise the Project.

Figure 14-1 presents the four principal zones for the Project.

Figure 14-1: Z87, J, SW, and X22 Zones; plan view



Source: AGP (2023)

The four principal zones were estimated separately using a mine grid coordinate system, rotated approximately 55° Az from the UTM coordinate NAD83 system. The mineral resource estimates used block matrices of 5 m x 5 m x 5 m. The blocks model grades were estimated using ordinary kriging interpolation method using 2 m (All Zones) capped composite values. Metal grades were capped prior to compositing. Capping levels vary based on mineralized domain where required. Density was assigned based on lithology models.

The Mineral Resources amenable to open pit extraction are reported within optimized constraining shells for each mineralized zone at a 0.3 g/t AuEQ cut-off grade. The Mineral Resources amenable to underground extraction are reported based on a 0.9 g/t AuEQ cut-off grade within grade shells consolidating contiguous blocks below the constraining shells.

The optimized constraining shells were developed by AGP using MineSight software and incorporates metal recovery, geotechnical parameters, and estimated costs for each mineralized zone. The mineral resources are classified as Indicated Resources or Inferred Resources in accordance with the CIM Definitions of Mineral Resources and Mineral Reserves (2014).

Table 14-1 presents the combined mineral resources, amenable to open pit and underground, for the Project.

Table 14-1: Combined Open Pit and Underground Resources

		Grade				Contained Metal		
Class	Tonnes (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	AuEQ (g/t	Au (Moz)	Cu (Mlbs)	Ag (Moz)
All Zones								
Indicated	508.3	0.57	0.07	1.09	0.69	9.32	729.50	17.79
Inferred	80.5	0.58	0.07	1.47	0.69	1.49	115.41	3.81

Notes:
Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
Summation errors may occur due to rounding.

AGP is not aware of any information not already discussed in this report, which would affect their interpretation or conclusions regarding the subject property. AGP is required to inform the public that the quantity and grade of reported Inferred resources in this estimation must be regarded as conceptual in nature and are based on limited geological evidence and sampling. The geological evidence is sufficient to imply, but not verify, geological grade or quality of continuity. For these reasons, an Inferred resource has a lower level of confidence than an Indicated resource. It is reasonably expected that most of the Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration. The rounding of values, as required by the reporting guidelines, may result in apparent differences between tonnes, grade, and metal content.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

14.1.1 Open Pit Resources

The mineral resources for the Project deposit amenable to open pit extraction at a 0.3 g/t AuEQ cut-off grade are: Indicated Resource of 506.2 Mt at 0.57 g/t Au, 0.07 %Cu, 1.09 g/t Ag and 0.68 g/t AuEQ;

and an Inferred Resource of 76.5 Mt at 0.53 g/t Au, 0.06 %Cu, 1.12 g/t Ag and 0.65 g/t AuEQ. Table 14-2 presents the Mineral Resources amenable to open pit extraction.

Table 14-2: Open Pit Mineral Resources for the Troilus Project at a 0.3 g/t AuEQ Cut-off Grade – All Zones

		Grade				Contained Metal			
Class	Tonnes (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	AuEQ (g/t)	Au (Moz)	Cu (Mlb)	Ag (g/t)	AuEQ (Moz)
Z87									
Indicated	197.1	0.67	0.07	1.21	0.80	4.21	320.69	7.67	5.04
Inferred	37.1	0.59	0.06	1.11	0.70	0.71	50.17	1.33	0.84
JZ									
Indicated	151.9	0.50	0.06	0.96	0.61	2.45	215.71	4.71	2.98
Inferred	24.2	0.46	0.07	0.94	0.57	0.35	35.37	0.73	0.44
X22									
Indicated	59.2	0.51	0.06	1.24	0.62	0.98	79.34	2.35	1.19
Inferred	13.6	0.53	0.07	1.48	0.67	0.23	21.76	0.65	0.29
SW									
Indicated	98.0	0.50	0.05	0.94	0.60	1.59	109.91	2.94	1.89
Inferred	1.6	0.37	0.04	0.96	0.45	0.02	1.36	0.05	0.02
TOTALS – ALL ZONES									
Indicated	506.2	0.57	0.07	1.09	0.68	9.23	725.66	17.67	11.11
Inferred	76.5	0.53	0.06	1.12	0.65	1.31	108.66	2.75	1.59

Notes:

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Summation errors may occur due to rounding.

Open pit mineral resources are reported within optimized constraining shells.

Open pit cut-off grade is 0.3 g/t AuEQ.

AuEQ equivalents were calculated as follows:

- Z87 Zone AuEQ = Au grade + 1.5628 * Cu grade + 0.0128 * Ag grade
- J Zone AuEQ = Au grade + 1.5107 * Cu grade + 0.0119 * Ag grade
- X22 Zone AuEQ = Au grade + 1.5628 * Cu grade + 0.0128* Ag grade
- SW Zone AuEQ = Au grade + 1.6823 * Cu grade + 0.0124 * Ag grade

Metal prices for the AuEQ formulas are: US$ 1,850/ oz Au; $4.25/lb Cu, and $23.00/ oz Ag; with an exchange rate of US$1.00: CAD$1.30

Metal recoveries for the AuEQ formulas are:

- Z87 Zone 95.5% for Au recovery, 94.7% for Cu recovery and 98.2% for Ag recovery
- J Zone 93.1% for Au recovery, 89.3% for Cu recovery and 88.9% for Ag recovery
- X22 Zone 95.5% for Au recovery, 94.7% for Cu recovery and 98.2% for Ag recovery
- SW Zone 85.7% for Au recovery, 91.5% for Cu recovery and 85.6% for Ag recovery

Capping of grades varied between 2.30 g/t Au and 21.00 g/t Au; between 0.06% Cu and 4.36 %Cu, and between 3.20 g/t Ag and 55.00 g/t Ag; on raw assays

The density (excluding overburden and fill) varies between 2.64 g/cm^3 and 2.93 g/cm^3 depending on lithology

14.2 Underground Resources

The mineral resources for the Project deposit amenable to underground extraction at a 0.9 g/t AuEQ cut-off grade are: an Indicated Resource of 2.1 Mt at 1.35 g/t Au, 0.09 %Cu, 1.90 g/t Ag and 1.51 g/t

AuEQ; and an Inferred Resource of 4.0 Mt at 1.36 g/t Au, 0.08 %Cu, 8.21 g/t Ag and 1.58 g/t AuEQ. Table 14-3 presents the Mineral Resources amenable to underground extraction.

Table 14-3: Underground Mineral Resources for the Troilus Project at a 0.9 g/t AuEQ Cut-off Grade – All Zones

		Grade				Contained Metal			
Class	Tonnes (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	AuEQ (g/t)	Au (Moz)	Cu (Mlb)	Ag (Moz)	AuEQ (Moz)
Z87									
Indicated	0.5	1.59	0.15	0.54	1.83	0.02	1.55	0.01	0.03
Inferred	1.1	1.99	0.12	0.46	2.19	0.07	2.96	0.02	0.08
JZ									
Indicated	0.2	1.21	0.07	1.46	1.33	0.01	0.29	0.01	0.01
Inferred	1.0	1.25	0.05	0.99	1.34	0.04	1.13	0.03	0.04
X22									
-none-									
-none-									
SW									
Indicated	1.4	1.28	0.07	2.44	1.42	0.06	2.00	0.11	0.06
Inferred	1.9	1.05	0.06	16.62	1.37	0.06	2.66	1.01	0.08
TOTALS – ALL ZONES									
Indicated	2.1	1.35	0.09	1.90	1.51	0.09	3.84	0.13	0.10
Inferred	4.0	1.36	0.08	8.21	1.58	0.18	6.75	1.06	0.20

Notes:

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability

Summation errors may occur due to rounding

Underground resources reported in 0.9 g/t AuEQ grade shells

Underground cut-off grade is 0.9 g/t AuEQ

AuEQ equivalents were calculated as follows:

Z87 Zone AuEQ = Au grade + 1.5628 * Cu grade + 0.0128 * Ag grade
J4/J5 Zone AuEQ = Au grade + 1.5107 * Cu grade + 0.0119 * Ag grade
X22 Zone AuEQ = Au grade + 1.5628 * Cu grade + 0.0128* Ag grade
SW Zone AuEQ = Au grade + 1.6823 * Cu grade + 0.0124 * Ag grade

Metal prices for the AuEQ formulas are: US$ 1,700/ oz Au; $4.25/lb Cu, and $23.00/ oz Ag; with an exchange rate of US$1.00: CAD$1.30

Metal recoveries for the AuEQ formulas are:

Z87 Zone 95.5% for Au recovery, 94.7% for Cu recovery and 98.2% for Ag recovery
J Zone 93.1% for Au recovery, 89.3% for Cu recovery and 88.9% for Ag recovery
X22 Zone 95.5% for Au recovery, 94.7% for Cu recovery and 98.2% for Ag recovery
SW Zone 85.7% for Au recovery, 91.5% for Cu recovery and 85.6% for Ag recovery

Capping of grades varied between 2.30 g/t Au and 21.00 g/t Au; between 0.06% Cu and 4.36 %Cu, and between 3.20 g/t Ag and 55.00 g/t Ag; on raw assays

The density (excluding overburden and fill) varies between 2.64 g/cm^3 and 2.93 g/cm^3 depending on lithology.

14.3 Database

The database for the Project contains all drilling on the Property. As of 31 August 2023, the Troilus drill hole database for Z87, J, and X22 Zones contains 1,492 surface diamond drill holes with a total length of 449,168 m. The database includes regional and exploration holes considered outside of the four principal deposits. Due to the advancement of drill programs in Z87, J and X22 Zones, several drill holes are shared between Zones. Table 14-4 presents a summary of drill holes in the database and drill holes used in the estimation of mineral resources for each Mineralized Zone.

Table 14-4: Drillhole Database Summary – Troilus Project

Mineralized Zone	Number of Drillholes	Length (m)	Comments
Z87 Zone	519	159,735	
J Zone	382	104,592	
Z87 Zone / J Zone			11 drill holes are shared
X22 Zone	174	66,634	
X22 Zone / Z87 Zone			61 drill holes are shared
SW Zone	320	130,597	

14.4 Lithological Model

14.4.1 Lithological Model

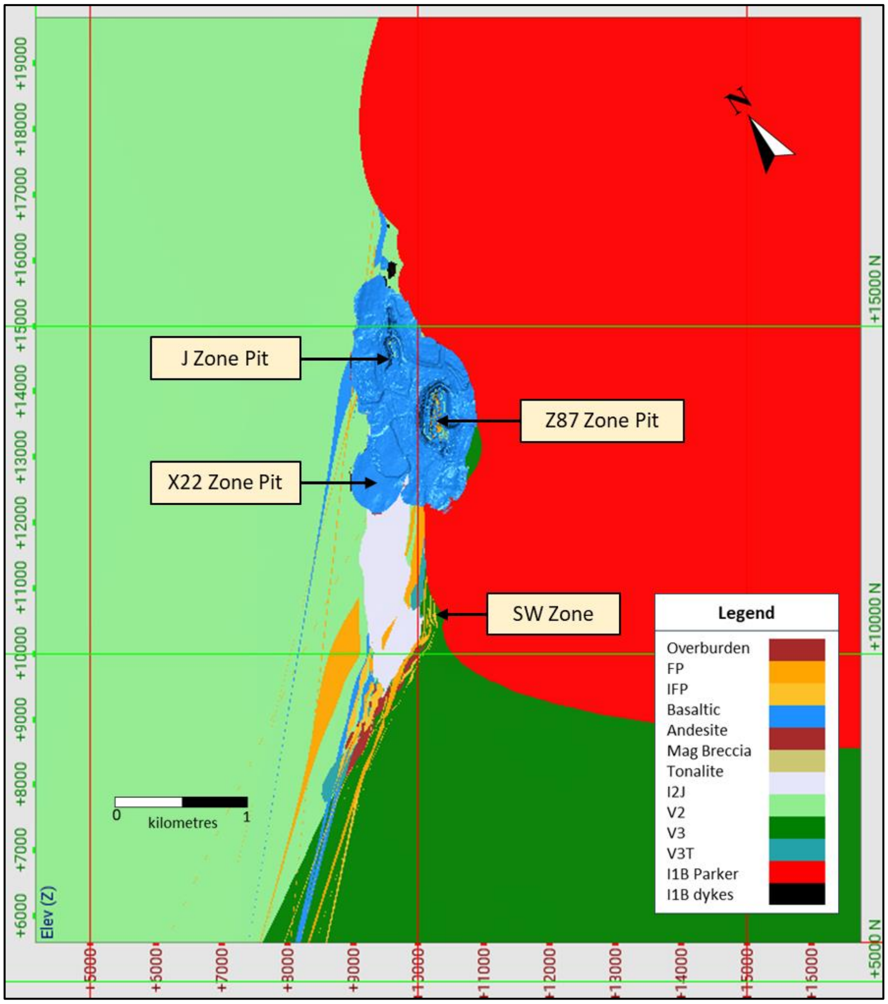
The three-dimensional wireframes model for the lithology model was interpreted by Troilus personnel using Leapfrog software. The lithology model is based on drill hole logs from all drill holes in the Project area. AGP has reviewed this model and has accepted the interpretation. The lithology model was used to code the lithology codes into the block model for the four mineralized zones and used to support the density model.

Table 14-5 presents the lithology codes used for the Project. Figure 14-2 presents a plan view of the lithology model, at 5200 m model elevation, for the four mineralized zones, with respect to the optimized pit constraint for the northern zones.

Table 14-5: Lithology Codes – Troilus Gold Project

Lithology	Lithology	Lithocode
Feldspar Porphyry	FP	61
Porphyritic Intrusive	IFP	62
Basaltic Andesite	Basaltic Andesite	63
Magnetite Breccia	Mag Breccia (only in SW)	64
Tonalite	Tonalite	65
Intermediate Volcanics	I2J	66
Mafic Volcanics (East)	V2	67
Mafic Volcanics (West)	V3	68
Mafic Volcanics (West)	V3T	69
Parker Granite	I1B Parker	70
Granitic Dykes	I1B_dykes	71
Overburden	OB	9

Figure 14-2: Lithology Model (Plan View at 5200 m Elevation); showing north deposits pit constraint

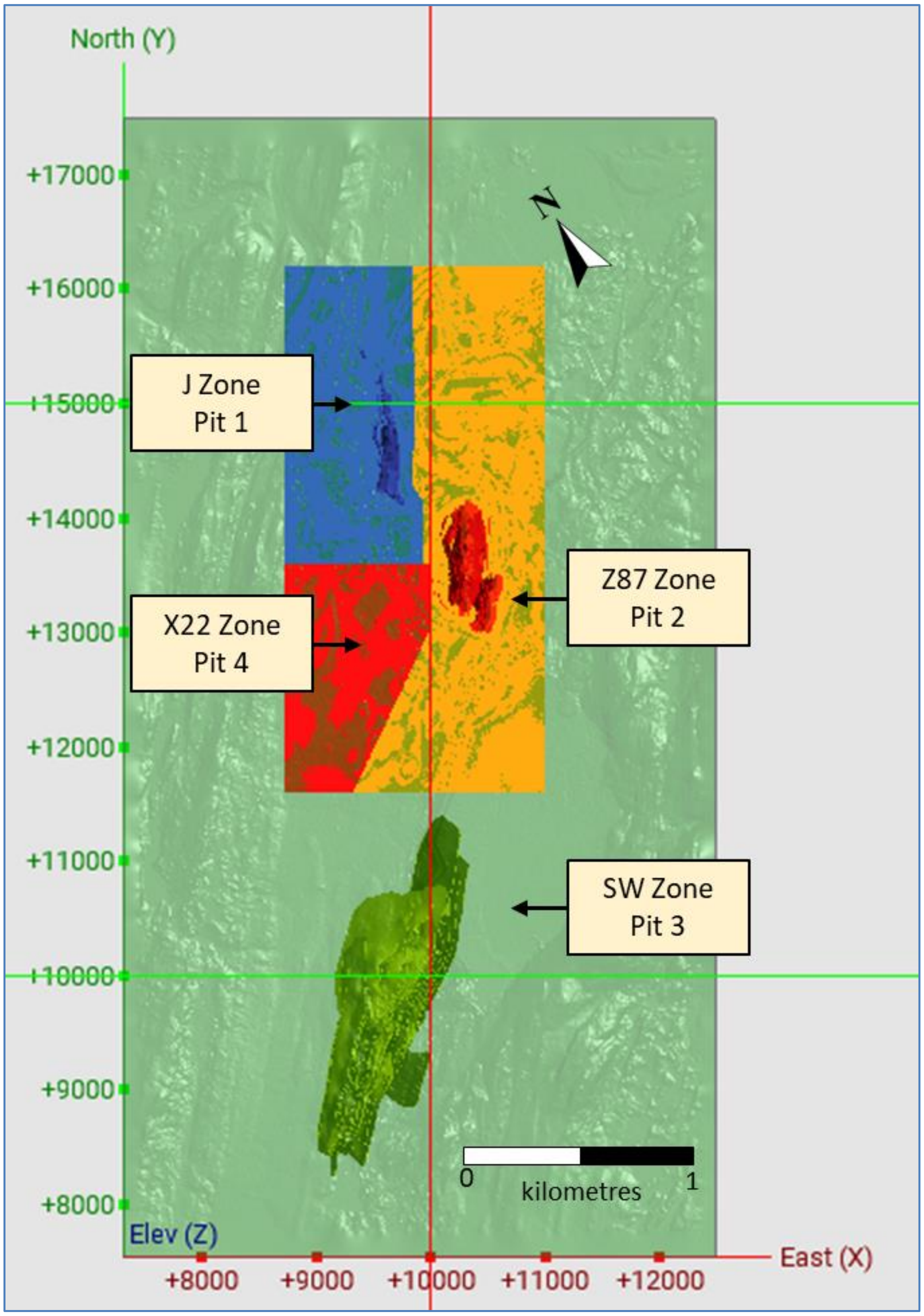


Source: AGP (2023)

14.4.2 Pit Areas

In order to differentiate between the three northern mineralized zones and avoid overlapping grade blocks within respective low-grade domains, a Pit code was populated in the models to keep the mineralized zones separated. The 'Pit Area' was coded into the block model as follows: J Zone =1, Z87 Zone = 2, SW Zone = 3 and X22 Zone = 4. Figure 14-3 shows the Pit Areas coded into the block models.

Figure 14-3: Pit Areas – Troilus Project; plan view



Source: AGP (2023)

14.4.3 Bulk Density

Z87, J and X22 Zones

A total of 132,983 density measurements were collected by Troilus from drill core during the 2019 - 2023 drill programs. The density assignment for the three northern mineralized zones is based on the mean density values within each lithology. Density for overburden was assigned the value of 2.20 g/cm^3.

Table 14-6 shows the densities assigned to each lithological domain in the SW Zone. Table 14-7 shows the descriptive statistics for the SW Zone by domain.

Table 14-6: Assigned Densities by Domain – Z87, J and X22 Zones

Lithology	Domain	Density
FP	61	2.73
IFP	62	2.82
Basaltic Andesite	63	2.80
Mag Breccia (in SW Zone only)	64	-
Tonalite	65	2.72
I2J	66	2.79
V2	67	2.76
V3	68	2.93
V3T	69	2.87
I1B Parker	70	2.72
I1B_dykes	71	2.64
Overburden	9	2.20

Table 14-7: Descriptive Statistics for Density by Lithology – Z87, J and X22 Zones

Lithology	Count	Min	Max	Mean	Median	StDev	CV
FP	16386	1.97	3.50	2.73	2.71	0.08	0.03
IFP	114	2.64	3.04	2.82	2.75	0.11	0.04
Basaltic Andesite	15597	2.19	3.47	2.80	2.79	0.08	0.03
Mag Breccia (only in SW)							
Tonalite	11619	2.24	3.61	2.72	2.72	0.05	0.02
I2J	22252	2.02	3.79	2.79	2.79	0.05	0.02
V2	61833	2.06	3.67	2.76	2.75	0.06	0.02
V3	996	2.61	3.13	2.93	2.94	0.09	0.03
V3T	1074	2.22	3.25	2.87	2.88	0.11	0.04
I1B Parker	967	2.53	3.11	2.72	2.65	0.14	0.05
I1B_dykes	2145	2.07	3.42	2.64	2.62	0.06	0.02

SW Zone

A total of 112,878 density measurements were collected by Troilus from drill core during the 2019 - 2023 drill programs in the SW Zone. The density assignment for the SW Zone is based on the mean density values within each lithology. Density for Overburden was assigned the value of 2.20.

Table 14-8 shows the densities assigned to each lithological domain in the SW Zone. Table 14-9 shows the descriptive statistics for the SW Zone by domain.

Table 14-8: Assigned Densities by Domain – SW Zone

Lithology	Lithocode	Density
FP	61	2.72
IFP	62	2.76
Basaltic Andesite	63	2.75
Mag Breccia	64	2.87
Tonalite (not in SW Zone)	65	-
I2J	66	2.80
V2	67	2.75
V3	68	2.92
V3T	69	2.82
I1B Parker	70	2.76
I1B_dykes (not in SW Zone)	71	-
Overburden	9	2.20

Table 14-9: Descriptive Statistics for Density by Lithology – SW Zone

Lithology	Count	Min	Max	Mean	Median	StDev	CV
FP	15336	2.14	3.41	2.72	2.7	0.08	0.03
IFP	13716	2.15	4.63	2.76	2.74	0.09	0.03
Basaltic Andesite	1898	2.51	3.12	2.75	2.75	0.06	0.02
Mag Breccia	13305	2.08	3.59	2.87	2.87	0.10	0.03
Tonalite (not in SW Zone)							
I2J	26858	2.26	3.71	2.80	2.8	0.06	0.02
V2	3129	2.43	3.26	2.75	2.75	0.07	0.02
V3	36157	1.81	3.93	2.92	2.94	0.10	0.03
V3T	2069	2.54	3.85	2.82	2.81	0.10	0.03
I1B Parker	410	2.43	3.24	2.76	2.75	0.12	0.04
I1B_dykes (not in SW Zone)							

14.5 Z87 Zone

14.5.1 Interpretation

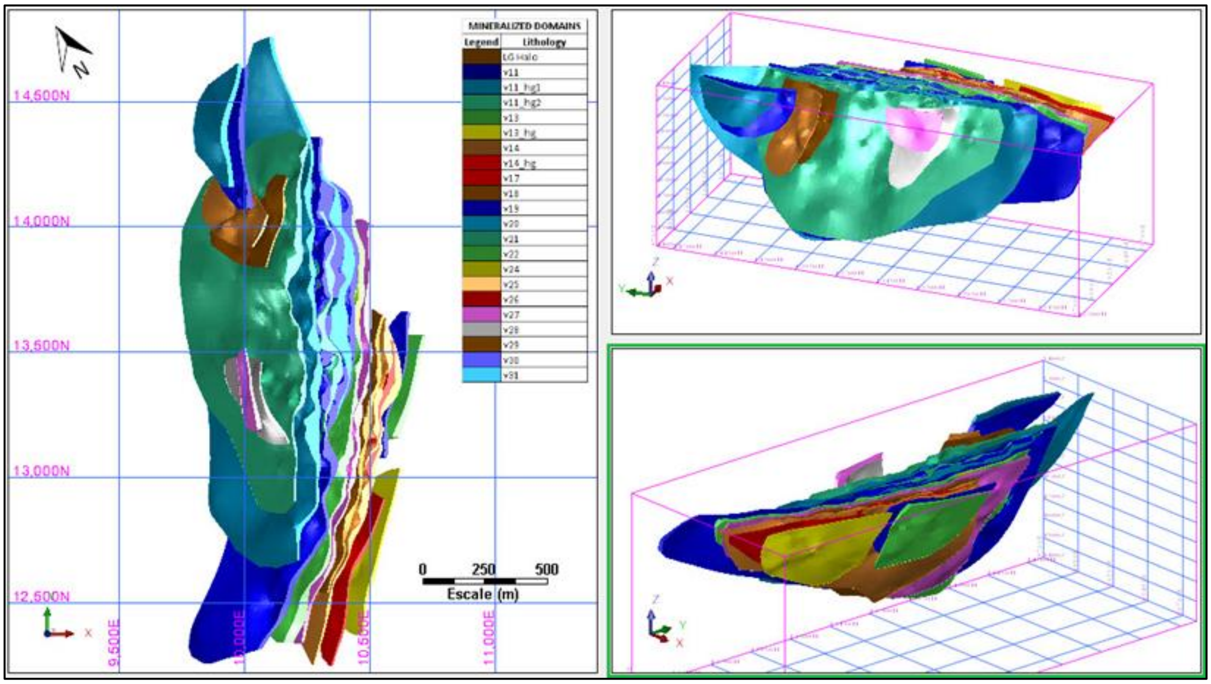
The three-dimensional (3D) wireframes models of the Z87 Zone were interpreted by Troilus personnel using Leapfrog™ Geo software, where grades were captured using a minimum grade of 0.3 g/t AuEQ, where a minimum thickness of 5 m was applied to all zones. Several higher-grade domains were captured using a minimum grade of 1.0 g/t AuEQ. The AuEQ formula used for the gold equivalent formula from the PEA Report (AGP, 2020a) as follows:

$$AuEQ = Au\ grade + (1.2566 * Cu\ grade) + (0.0103 * Ag\ grade)$$

All mineralized domain envelopes were created above pre-mining topography and clipped to the overburden bottom surface and then, to the pit topography. A total of 22 3D wireframes describes the mineralized domains in Z87 Zone. This includes a surrounding low-grade domain (or halo), created based on a grade greater than 0.07 g/t Au. The mineralization is disseminated and shows enrichment in what could be described as mineralized corridors without sharp boundaries. AGP considers the wireframes suitable to estimate resources. All wireframes were imported and validated into Surpac™ software.

Figure 14-4 shows the mineralized domain wireframes for the Z87 Zone. Table 14-10 summarizes the mineralized domains and domain codes.

Figure 14-4: Mineralized Zones – Z87 Zone



Source: AGP (2023)

Table 14-10: Mineralized Zones and Domain Codes – Z87 Zone

Mineralized Zone	Domain Code	Mineralized Zone	Domain Code
HG Halo	3100	v20	3200
v11	3110	v21	3210
v11_hg1	3115	v22	3220
v11_hg2	3120	v24	3240
v13	3130	v25	3250
v13_hg	3135	v26	3260
v14	3140	v27	3270
v14_hg	3145	v28	3280
v17	3170	v29	3290
v18	3180	v30	3300
v19	3190	v31	3310

14.5.2 Exploratory Data Analysis

Raw Assays

The drill hole database for the mineralized domains in the Z87 Zone, consists of 95,644 raw assay values for each metal: gold, copper, and silver. The assay values reported below detection limit were assigned half the detection limit for statistical analysis and grade estimation. Any missing values were assigned half the detection limit.

Table 14-11 to Table 14-13 presents the descriptive statistics for gold, copper, and silver, respectively, by mineralized domain in Z87 Zone.

Table 14-11: Descriptive Statistics for Au (g/t) Assays by Mineralized Domain – Z87 Zone

Mineralized Zone	Count	Min	Max	Mean	Median	StDev	CV
LG Halo	54,773	0.001	67.40	0.15	0.05	0.68	4.45
v11	10,439	0.001	154.00	0.43	0.21	1.72	4.05
v11 HG-1	6,843	0.001	103.01	1.48	0.82	3.17	2.14
v11 HG-2	2,834	0.001	23.51	1.00	0.67	1.26	1.26
v13	3,411	0.001	133.70	0.54	0.21	2.73	5.07
v13 HG	737	0.001	108.16	1.36	0.37	5.55	4.07
v14	2,878	0.001	25.94	0.46	0.22	1.13	2.46
v14 HG	1,132	0.001	87.40	1.18	0.53	3.51	2.99
v17	2,949	0.001	24.80	0.51	0.24	1.16	2.28
v18	1,750	0.001	29.07	0.47	0.19	1.25	2.68
v19	479	0.001	17.68	0.37	0.14	0.97	2.62
v20	3,537	0.001	26.00	0.42	0.22	0.85	2.02
v21	1,978	0.001	10.96	0.31	0.15	0.57	1.82
v22	185	0.002	11.45	0.47	0.17	1.11	2.38
v24	200	0.003	13.15	0.41	0.16	1.11	2.74
v25	235	0.003	5.39	0.32	0.17	0.56	1.74
v26	242	0.003	7.08	0.31	0.14	0.64	2.07
v27	123	0.003	15.35	0.57	0.19	1.51	2.66
v28	214	0.003	17.55	0.46	0.19	1.30	2.85
v29	239	0.003	3.81	0.26	0.14	0.42	1.64
v30	316	0.001	61.98	0.57	0.13	3.57	6.33
v31	150	0.001	5.24	0.34	0.19	0.62	1.82

Table 14-12: Descriptive Statistics for Cu (%) Assays by Mineralized Domain – Z87 Zone

Mineralized Zone	Count	Min	Max	Mean	Median	StDev	CV
LG Halo	54,773	-	30.00	0.02	0.01	0.13	6.31
v11	10,439	-	3.33	0.05	0.03	0.09	1.59
v11 HG-1	6,843	-	10.00	0.17	0.10	0.30	1.72
v11 HG-2	2,834	-	1.02	0.09	0.05	0.10	1.18
v13	3,411	-	11.27	0.07	0.03	0.24	3.47
v13 HG	737	0.001	1.74	0.11	0.04	0.18	1.71
v14	2,878	-	1.17	0.05	0.02	0.08	1.71
v14 HG	1,132	0.001	0.54	0.06	0.04	0.06	1.13
v17	2,949	-	2.926	0.06	0.02	0.14	2.33
v18	1,750	-	0.538	0.03	0.02	0.04	1.37
v19	479	0.001	2.00	0.02	0.01	0.10	3.94
v20	3,537	-	0.78	0.04	0.02	0.07	1.51
v21	1,978	-	1.26	0.04	0.02	0.07	1.86
v22	185	0.001	0.23	0.02	0.01	0.03	1.59
v24	200	0.001	0.43	0.04	0.03	0.04	1.14
v25	235	0.001	1.12	0.08	0.06	0.10	1.27
v26	242	0.001	0.88	0.05	0.03	0.08	1.59
v27	123	0.001	0.39	0.02	0.01	0.05	2.43
v28	214	-	0.33	0.03	0.01	0.05	1.64
v29	239	-	0.70	0.06	0.03	0.08	1.43
v30	316	0.001	2.12	0.10	0.05	0.19	1.79
v31	150	-	1.96	0.05	0.02	0.17	3.28

Table 14-13: Descriptive Statistics for Ag (g/t) Assays by Mineralized Domain – Z87 Zone

Mineralized Zone	Count	Min	Max	Mean	Median	StDev	CV
LG Halo	54,773	-	720.00	0.54	0.25	3.30	6.07
v11	10,439	-	48.00	0.86	0.30	1.48	1.72
v11 HG-1	6,843	-	145.00	1.86	1.00	4.73	2.55
v11 HG-2	2,834	-	41.00	0.96	0.40	1.98	2.06
v13	3,411	-	91.90	0.99	0.25	3.19	3.23
v13 HG	737	-	34.60	1.33	0.30	3.04	2.28
v14	2,878	-	95.11	1.11	0.50	2.69	2.43
v14 HG	1,132	-	159.5	1.76	0.90	5.33	3.03
v17	2,949	-	54.00	0.98	0.40	2.06	2.11
v18	1,750	-	23.45	0.73	0.30	1.25	1.71
v19	479	-	39.20	0.97	0.50	2.52	2.61
v20	3,537	-	259.90	1.07	0.30	6.39	6.00
v21	1,978	-	32.80	0.71	0.25	1.67	2.35
v22	185	0.10	6.20	0.78	0.40	1.05	1.33
v24	200	0.10	51.20	2.09	1.10	4.37	2.09
v25	235	0.10	6.60	0.83	0.50	0.91	1.10
v26	242	0.10	471.00	3.42	0.70	30.31	8.87
v27	123	0.10	15.66	0.82	0.25	2.19	2.67
v28	214	0.10	5.80	0.61	0.25	0.72	1.19
v29	239	-	11.90	0.87	0.50	1.21	1.39
v30	316	0.10	67.90	1.65	0.70	4.60	2.78
v31	150	0.10	29.60	0.93	0.30	2.62	2.81

In the database, for the Z87 Zone, approximately 0.83% of the gold assays are missing a silver assay. These are largely associated with the pre-2018 drilling, ('KN-' drillholes). The other missing intervals were filled with half of the detection limit.

The matrix correlation for the combined low-grade and high-grade mineralized zones, indicated that gold correlates better with copper than silver although the correlation coefficient (R) is low (0.29 versus 0.24). Silver and copper show the best correlation with a R of 0.46. Despite the low correlation coefficient, AGP attempted a simple regression between copper and silver assays and as expected, the regression proved to be very poor, precluding the calculation of the missing silver assays. It was therefore decided that all missing assays and zero results will be set to zero grade during the compositing process. Table 14-14 shows the correlation matrix between gold, copper, and silver.

Table 14-14: Correlation Matrix – Z87 Zone

Column	Au (g/t)	Ag (g/t)	Cu (%)
Au (g/t)	1	0.24	0.29
Ag (g/t)	0.24	1	0.46
Cu (%)	0.29	0.46	1

Capping Analysis

The individual mineralized zones were grouped into six domains (Grouped Domains) to perform the capping analysis. Table 14-15 shows the domains, and the associated mineralized zones.

Table 14-15: Grouped Domains and Mineralized Zones within Grouped Domains – Z87 Zone

Domain 1	Domain 2	Domain 3	Domain 4	Domain 5	Domain 6
LG Halo	v11	v14	v27	v25	v30
	v11 HG-1	v14 HG	v28	v29	v31
	v11 HG-2	v17			
	v13	v18			
	v13 HG	v19			
	v20	v22			
	v21	v24			
		v26			

The coefficient of variation is considered low for a gold system and therefore aggressive outlier control was not deemed necessary. Capping levels were determine using probability plots and decile analysis (Parish, 1997). Capping of raw assays typically affected the upper 99.3 percentile (or above) of the population. Table 14-16 presents the selected gold, silver, and copper capping levels by domain.

Table 14-16: Capping Levels by Grouped Domain – Z87 Zone

Domain	Au (g/t)	Loss (%)	Cu (%)	Loss (%)	Ag (g/t)	Loss (%)
Domain 01	2.90 (236)	0.01	1.23 (4)	0.03	31.13 (11)	0.03
Domain 02	21.00 (49)	0.04	4.36 (4)	0.01	32.00 (42)	0.04
Domain 03	12.00 (23)	0.05	2.00 (2)	0.003	39.20 (6)	0.06
Domain 04	4.10 (2)	0.15	0.24 (3)	0.04	11.5 (2)	0.03
Domain 05	3.91 (1)	0.01	0.70 (1)	0.01	6.60 (1)	0.01
Domain 06	5.24 (2)	0.28	1.36 (2)	0.03	25. (2)	0.07

(x) – number of values capped

Table 14-17 to Table 14-19 present the descriptive statistics for capped gold, copper, and silver assay values by Grouped Domain, respectively.

Table 14-17: Descriptive Statistics for Capped Au (g/t) Assays by Mineralized Zone – Z87 Zone

Mineralized Zone	Count	Min	Max	Mean	Median	StDev	CV
LG Halo	54,773	0	2.90	0.13	0.05	0.09	0.30
v11	10,439	0	21.00	0.41	0.21	0.70	0.84
v11 HG-1	6,843	0	21.00	1.41	0.82	4.81	2.19
v11 HG-2	2,834	0.001	21.00	1.00	0.67	1.55	1.25
v13	3,411	0	21.00	0.49	0.21	1.65	1.28
v13 HG	737	0.001	21.00	1.10	0.37	6.80	2.61
v14	2,878	0	12.00	0.45	0.22	0.83	0.91
v14 HG	1,132	0	12.00	1.05	0.53	2.57	1.60
v17	2,949	0	12.00	0.49	0.24	0.95	0.97
v18	1,750	0	12.00	0.45	0.19	0.93	0.97
v19	479	0.001	12.00	0.36	0.14	0.59	0.77
v20	3,537	0	21.00	0.42	0.22	0.66	0.81
v21	1,978	0.001	10.96	0.31	0.15	0.32	0.57
v22	185	0.002	11.45	0.47	0.17	1.23	1.11
v24	201	0	12.00	0.40	0.16	1.09	1.04
v25	235	0.003	3.91	0.31	0.17	0.26	0.51
v26	242	0.003	7.08	0.31	0.14	0.41	0.64
v27	123	0.003	4.10	0.48	0.19	0.59	0.77
v28	214	0.003	4.10	0.39	0.19	0.39	0.62
v29	239	0	3.81	0.25	0.14	0.18	0.42
v30	316	0.001	5.24	0.36	0.13	0.49	0.70
v31	150	0.001	5.24	0.34	0.19	0.39	0.62

Table 14-18: Descriptive Statistics for Capped Cu (%) Assays by Mineralized Zone – Z87 Zone

Mineralized Zone	Count	Min	Max	Mean	Median	StDev	CV
LG Halo	54,773	0	1.23	0.02	0.01	0.00	0.04
v11	10,439	0	3.33	0.05	0.03	0.01	0.09
v11 HG-1	6,843	0	4.36	0.17	0.10	0.06	0.24
v11 HG-2	2,834	0	1.02	0.09	0.05	0.01	0.10
v13	3,411	0	4.36	0.07	0.03	0.03	0.17
v13 HG	737	0	1.74	0.11	0.04	0.03	0.18
v14	2,878	0	1.17	0.04	0.02	0.01	0.08
v14 HG	1,132	0	0.54	0.06	0.04	0.00	0.06
v17	2,949	0	2.00	0.06	0.02	0.02	0.13
v18	1,750	0	0.54	0.03	0.02	0.00	0.04
v19	479	0	2.00	0.02	0.01	0.01	0.09
v20	3,537	0	0.78	0.04	0.02	0.00	0.07
v21	1,978	0	1.26	0.04	0.02	0.01	0.07
v22	185	0	0.23	0.02	0.01	0.00	0.03
v24	201	0	0.43	0.04	0.03	0.00	0.04
v25	235	0	0.70	0.08	0.06	0.01	0.09
v26	242	0	0.88	0.05	0.03	0.01	0.08
v27	123	0	0.24	0.02	0.01	0.00	0.04
v28	214	0	0.24	0.03	0.01	0.00	0.04
v29	239	0	0.70	0.06	0.03	0.01	0.08
v30	316	0	1.36	0.10	0.05	0.03	0.16
v31	150	0	1.36	0.05	0.02	0.02	0.13

Table 14-19: Descriptive Statistics for Capped Ag (g/t) Assays by Mineralized Zone – Z87 Zone

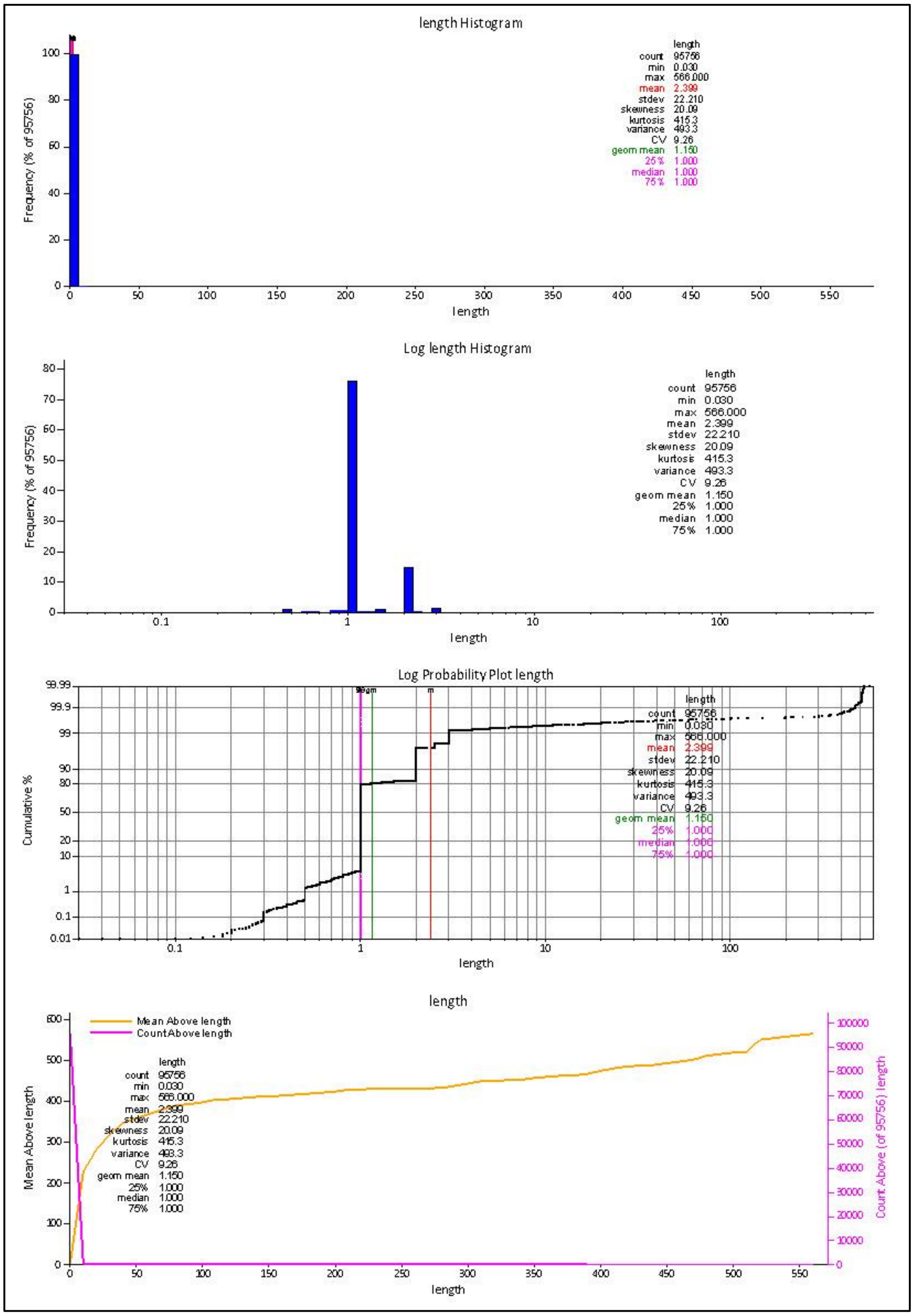
Mineralized Zone	Count	Min	Max	Mean	Median	StDev	CV
LG Halo	54,773	0	31.13	0.53	0.25	1.07	1.03
v11	10,439	0	32.00	0.86	0.30	2.08	1.44
v11 HG-1	6,843	0	32.00	1.75	1.00	8.80	2.97
v11 HG-2	2,834	0	32.00	0.96	0.40	3.48	1.87
v13	3,411	0	32.00	0.94	0.25	4.75	2.18
v13 HG	737	0	32.00	1.33	0.30	8.91	2.99
v14	2,878	0	39.20	1.08	0.50	4.27	2.07
v14 HG	1,132	0	39.20	1.65	0.90	7.59	2.75
v17	2,949	0	39.20	0.97	0.40	3.78	1.94
v18	1,750	0	23.45	0.73	0.30	1.55	1.25
v19	479	0	39.20	0.97	0.50	6.34	2.52
v20	3,537	0	32.00	0.93	0.30	4.12	2.03
v21	1,978	0	32.00	0.71	0.25	2.77	1.66
v22	185	0.10	6.20	0.78	0.40	1.09	1.05
v24	200	0	39.20	2.02	1.10	13.82	3.72
v25	235	0.10	6.60	0.83	0.50	0.83	0.91
v26	242	0.10	39.20	1.63	0.70	13.87	3.72
v27	123	0.10	11.50	0.76	0.25	3.32	1.82
v28	214	0.10	5.80	0.61	0.25	0.52	0.72
v29	239	0	6.60	0.84	0.50	1.09	1.04
v30	316	0.10	25.70	1.52	0.70	9.03	3.00
v31	150	0.10	25.70	0.91	0.30	5.46	2.34

Composites

Drill core was sampled mostly in 1 m or 2 m intervals; with the median sample length is 1.0 m. AGP elected to composite the data in 2.0 m intervals. Composites were created within the mineralized domains, starting from the domain intersection. Composite lengths were adjusted equally across the domain intersection.

Figure 14-5 shows the statistics of the raw assays lengths used to determine the composite interval.

Figure 14-5: Statistical Analysis Plots to Determine Composite Interval – Z87 Zone



Source: AGP (2023)

Table 14-20 to Table 14-22 present the descriptive statistics for the 2 m capped composite values for gold, copper, and silver by domain, respectively, in the Z87 Zone.

Table 14-20: Descriptive Statistics for Capped Au (g/t) Composite Values by Mineralized Domain – Z87 Zone

Zone	Count	Min	Max	Mean	Median	StDev	CV
Total	64,802	0	23.65	0.32	0.09	0.75	2.34
LG Halo	38,706	0	2.895	0.11	0.05	0.23	2.03
v11	6,452	0	23.65	0.41	0.26	0.66	1.63
v11 HG-1	4,240	0	21.00	1.39	0.94	1.68	1.21
v11 HG -2	1,759	0.003	15.05	0.98	0.75	1.02	1.04
v13	2,237	0	21.00	0.49	0.25	1.15	2.34
v13 HG	450	0.001	21.00	1.0949	0.49	1.97	1.80
v14	1,823	0	7.85	0.4279	0.25	0.67	1.56
v14 HG	662	0	12.00	1.03	0.67	1.24	1.20
v17	1,949	0	12.00	0.4929	0.28	0.82	1.66
v18	1,173	0	12.00	0.4264	0.23	0.75	1.76
v19	294	0.001	6.11	0.3572	0.20	0.59	1.65
v20	2,284	0	10.52	0.39	0.24	0.62	1.57
v21	1,428	0.003	3.90	0.26	0.14	0.40	1.50
v22	121	0.003	5.78	0.44	0.19	0.80	1.80
v24	157	0	6.07	0.38	0.17	0.76	1.99
v25	161	0.003	2.90	0.24	0.14	0.38	1.57
v26	172	0.005	3.83	0.290	0.17	0.42	1.44
v27	87	0.003	4.10	0.44	0.16	0.75	1.69
v28	193	0.003	4.10	0.27	0.09	0.49	1.82
v29	202	0	1.97	0.17	0.08	0.29	1.72
v30	174	0.007	3.39	0.36	0.16	0.56	1.55
v31	78	0.012	2.77	0.33	0.21	0.44	1.34

Table 14-21: Descriptive Statistics for Capped Cu (%) Composite Values by Mineralized Domain – Z87 Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
	64,802	0	3.72	0.04	0.01	0.08	2.07
lg_halo	38,706	0	0.7294	0.02	0.01	0.03	1.83
v11	6,452	0	1.23	0.05	0.03	0.07	1.27
v11_hg1	4,240	0	2.86	0.17	0.11	0.18	1.08
v11_hg2	1,759	0.0005	0.70	0.08	0.06	0.09	1.05
v13	2,237	0	3.72	0.06	0.03	0.13	2.09
v13_hg	450	0.0005	1.11	0.10	0.04	0.15	1.44
v14	1,823	0	0.71	0.04	0.03	0.06	1.43
v14_hg	662	0	0.46	0.06	0.04	0.06	0.98
v17	1,949	0	2.00	0.06	0.03	0.11	1.95
v18	1,173	0	0.54	0.03	0.02	0.04	1.29
v19	294	0.0005	0.33	0.02	0.01	0.03	1.48
v20	2,284	0	0.45	0.04	0.02	0.05	1.36
v21	1,428	0.0001	0.60	0.03	0.01	0.05	1.67
v22	121	0.0005	0.23	0.02	0.01	0.03	1.56
v24	157	0	0.22	0.04	0.03	0.03	0.87
v25	161	0.0005	0.42	0.06	0.04	0.07	1.18
v26	172	0.0023	0.88	0.06	0.03	0.08	1.50
v27	87	0.0005	0.24	0.02	0.01	0.04	2.21
v28	193	0.0005	0.14	0.02	0.01	0.03	1.64
v29	202	0	0.60	0.04	0.02	0.07	1.74
v30	174	0.0025	0.96	0.10	0.06	0.13	1.29
v31	78	0.0010	0.33	0.04	0.02	0.06	1.44

Table 14-22: Descriptive Statistics for Capped Ag (g/t) Composite Values by Mineralized Domain – Z87 Zone

Zone	Count	Min	Max	Mean	Median	StDev	CV
Total	103,508	0	39.20	0.49	0.25	1.09	2.21
lg_halo	77,412	0	31.13	0.29	0.13	0.58	1.96
v11	6,452	0	18.50	0.88	0.50	1.13	1.29
v11_hg1	4,240	0	32.00	1.71	1.18	2.17	1.27
v11_hg2	1,759	0	32.00	1.01	0.56	1.68	1.65
v13	2,237	0	32.00	0.95	0.45	1.88	1.97
v13_hg	450	0	28.70	1.29	0.50	2.29	1.78
v14	1,823	0	39.20	1.24	0.70	2.42	1.95
v14_hg	662	0	39.20	1.92	1.05	3.30	1.72
v17	1,949	0	29.98	1.01	0.55	1.67	1.65
v18	1,173	0	23.45	0.76	0.43	1.25	1.64
v19	294	0	17.97	1.00	0.62	1.52	1.53
v20	2,284	0	32.00	0.87	0.38	1.67	1.91
v21	1,428	0	13.69	0.59	0.25	1.09	1.84
v22	121	0.10	5.50	0.83	0.50	0.93	1.12
v24	157	0	19.73	1.77	1.00	2.38	1.34
v25	161	0.10	3.75	0.65	0.48	0.68	1.04
v26	172	0.15	29.40	1.71	0.90	3.25	1.90
v27	87	0.10	11.50	0.83	0.25	1.93	2.31
v28	193	0.10	3.60	0.47	0.25	0.56	1.19
v29	202	0	5.80	0.61	0.25	0.84	1.37
v30	174	0.10	15.84	1.41	0.71	2.16	1.54
v31	78	0.10	7.38	0.76	0.43	1.05	1.39

Variography

Variograms based on AGP composite files were generated in Surpac™ Variogram Modeling Windows module by the AGP personnel, which uses bearing, plunge, and dip to describe the anisotropy.

Table 14-23 to Table 14-25 show the variogram parameters used for interpolation. Gold, copper, and silver variograms were completed independently for each one of the domains.

Table 14-23: Gold Variogram Parameters – Z87 Zone

Zone	Anisotropy	Structure	Nugget	1st Sill	1st Range	2nd Sill	2nd Range	3rd Sill	3rd Range
LG Halo	Spherical	2	0.02	0.02	32.71	0.01	115.69	-	-
v11	Spherical	2	0.02	0.02	32.71	0.01	115.69	-	-
v11 HG-1	Spherical	2	0.02	0.02	32.71	0.01	115.69	-	-
v11 HG-2	Spherical	2	0.02	0.02	32.71	0.01	115.69	-	-
v13	Spherical	2	0.02	0.02	32.71	0.01	115.69	-	-
v13 HG	Spherical	2	0.02	0.02	32.71	0.01	115.69	-	-
v14	Spherical	2	0.02	0.02	32.71	0.01	115.69	-	-
v14 HG	Spherical	3	0.92	0.37	22.65	0.38	32.71	0.09	81.23
v17	Spherical	3	0.92	0.37	22.65	0.38	32.71	0.09	81.23
v18	Spherical	2	0.05	0.10	83.38	0.20	152.07	-	-
v19	Spherical	2	0.05	0.10	83.38	0.20	152.07	-	-
v20	Spherical	2	0.02	0.02	32.71	0.01	115.69	-	-
v21	Spherical	2	0.02	0.02	32.71	0.01	115.69	-	-
v22	Spherical	2	0.05	0.10	83.38	0.20	152.07	-	-
v24	Spherical	2	0.05	0.10	83.38	0.20	152.07	-	-
v25	Spherical	2	0.06	0.02	67.11	0.03	130.48	-	-
v26	Spherical	2	0.06	0.02	67.11	0.03	130.48	-	-
v27	Spherical	2	0.06	0.02	67.11	0.03	130.48	-	-
v28	Spherical	2	0.06	0.02	67.11	0.03	130.48	-	-
v29	Spherical	2	0.06	0.02	67.11	0.03	130.48	-	-
v30	Spherical	2	0.01	0.09	89.48	0.18	216.49	-	-
v31	Spherical	2	0.01	0.09	89.48	0.18	216.49	-	-

Table 14-24: Copper Variogram Parameters – Z87 Zone

Zone	Anisotropy	Structure	Nugget	1st Sill	1st Range	2nd Sill	2nd Range	3rd Sill	3rd Range
LG Halo	Spherical	2	0.0002	0.0002	62.67	0.0005	145.60	-	-
v11	Spherical	2	0.0002	0.0002	62.67	0.0005	145.60	-	-
v11 HG-1	Spherical	2	0.0002	0.0002	62.67	0.0005	145.60	-	-
v11 HG-2	Spherical	2	0.0002	0.0002	62.67	0.0005	145.60	-	-
v13	Spherical	2	0.0002	0.0002	62.67	0.0005	145.60	-	-
v13 HG	Spherical	2	0.0002	0.0002	62.67	0.0005	145.60	-	-
v20	Spherical	2	0.0002	0.0002	62.67	0.0005	145.60	-	-
v21	Spherical	2	0.0002	0.0002	62.67	0.0005	145.60	-	-
v14	Spherical	2	0.0002	0.0002	62.67	0.0005	145.60	-	-
v14 HG	Spherical	2	0.0070	0.0105	19.40	0.0028	100.00	-	-
v17	Spherical	2	0.0070	0.0105	19.40	0.0028	100.00	-	-
v18	Spherical	2	0.0002	0.0004	23.33	0.0005	97.78	-	-
v19	Spherical	2	0.0002	0.0004	23.33	0.0005	97.78	-	-
v22	Spherical	2	0.0002	0.0004	23.33	0.0005	97.78	-	-
v24	Spherical	2	0.0002	0.0004	23.33	0.0005	97.78	-	-
v26	Spherical	2	0.0005	0.0023	60.00	0.0020	140.00	-	-
v27	Spherical	2	0.0005	0.0023	60.00	0.0020	140.00	-	-
v28	Spherical	2	0.0005	0.0023	60.00	0.0020	140.00	-	-
v25	Spherical	2	0.0005	0.0023	60.00	0.0020	140.00	-	-
v29	Spherical	2	0.0005	0.0023	60.00	0.0020	140.00	-	-
v30	Spherical	2	0.0016	0.0048	69.88	0.0064	101.27	-	-
v31	Spherical	2	0.0016	0.0048	69.88	0.0064	101.27	-	-

Table 14-25: Silver Variogram Parameters – Z87 Zone

Zone	Anisotropy	Structure	Nugget	1st Sill	1st Range	2nd Sill	2nd Range	3rd Sill	3rd Range
LG Halo	Spherical	2	0.03	0.15	70.00	0.37	130.00	-	-
v11	Spherical	2	0.03	0.15	70.00	0.37	130.00	-	-
v11 HG-1	Spherical	2	0.03	0.15	70.00	0.37	130.00	-	-
v11 HG-2	Spherical	2	0.03	0.15	70.00	0.37	130.00	-	-
v13	Spherical	2	0.03	0.15	70.00	0.37	130.00	-	-
v13 HG	Spherical	2	0.03	0.15	70.00	0.37	130.00	-	-
v20	Spherical	2	0.03	0.15	70.00	0.37	130.00	-	-
v21	Spherical	2	0.03	0.15	70.00	0.37	130.00	-	-
v14	Spherical	2	0.03	0.15	70.00	0.37	130.00	-	-
v14 HG	Spherical	2	1.35	0.78	55.00	1.55	135.00	-	-
v17	Spherical	2	1.35	0.78	55.00	1.55	135.00	-	-
v18	Spherical	2	0.50	0.10	56.05	0.79	165.28	-	-
v19	Spherical	2	0.50	0.10	56.05	0.79	165.28	-	-
v22	Spherical	2	0.50	0.10	56.05	0.79	165.28	-	-
v24	Spherical	2	0.50	0.10	56.05	0.79	165.28	-	-
v26	Spherical	2	0.15	0.16	68.83	0.28	123.09	-	-
v27	Spherical	2	0.15	0.16	68.83	0.28	123.09	-	-
v28	Spherical	2	0.15	0.16	68.83	0.28	123.09	-	-
v25	Spherical	2	0.15	0.16	68.83	0.28	123.09	-	-
v29	Spherical	2	0.15	0.16	68.83	0.28	123.09	-	-
v30	Spherical	2	0.26	0.90	125.00	2.49	220.00	-	-
v31	Spherical	2	0.26	0.90	125.00	2.49	220.00	-	-

14.5.3 Block Model

Block Model Parameters

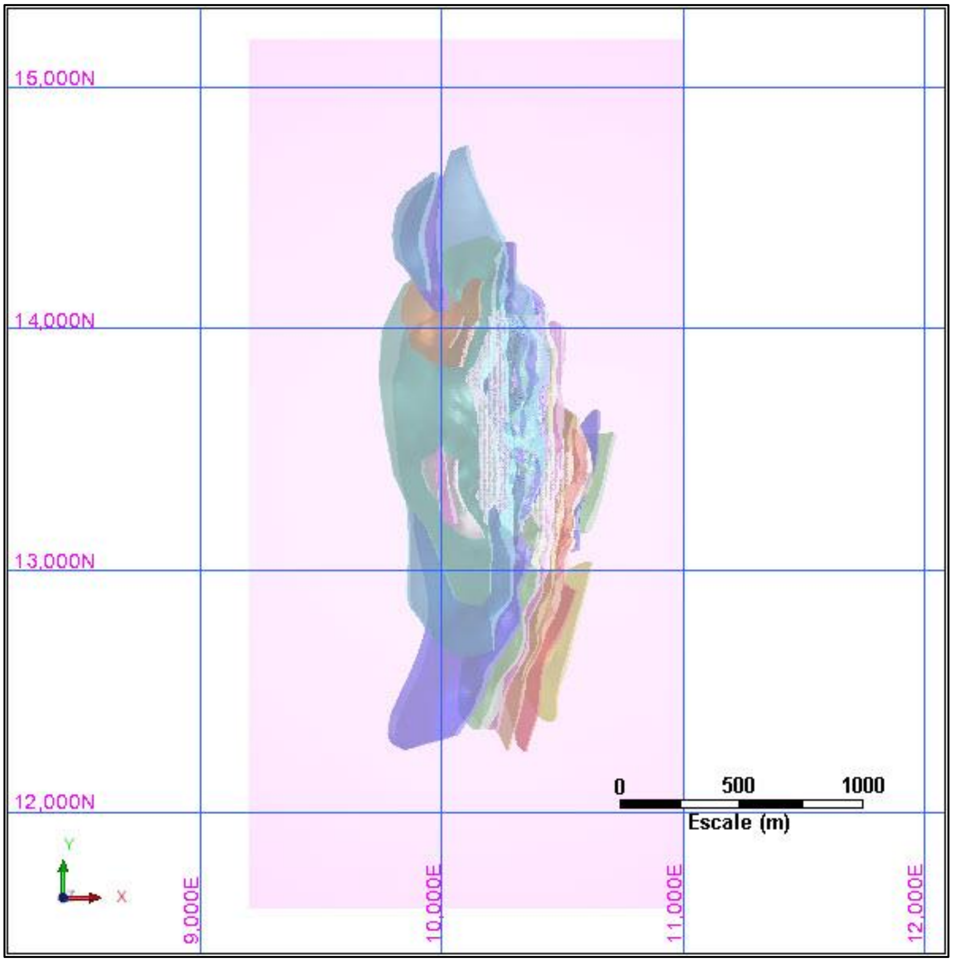
The block model for the Z87 Zone deposit was set up with a block matrix of 5 m long by 5 m wide by 5 m high and was built in Geovia Surpac resource software. The block matrix was defined based on current drill hole spacing and on engineering considerations for an open pit operation and is considered suitable this purpose. The block model is in mine grid coordinates and is not rotated. The block model is a whole block model where blocks are assigned a specific rock type code. Any block centroid within the mineralized domain wireframe was assigned that code.

Table 14-26 summarizes the block model parameters. Figure 14-6 presents the block model extents for the Z87 Zone.

Table 14-26: Block Model Parameters – Z87 Zone

Type	Y	X	Z
Minimum Coordinates	11600 mN	9200 mE	4500
Maximum Coordinates	15200 mN	11000 mE	5500
Length (m)	3600	1800	1000
Block Size (m)	5	5	5
Rotation	No rotation		
Number of Blocks	720	360	200

Figure 14-6: Block Model Extents – Z87 Zone



Source: AGP (2023)

Estimation Parameters and Interpolation Strategy

The metal grades were interpolated in three passes using the 2 m capped composites and using Geovia Surpac Dynamic Anisotropy.

The metal grades were interpolated using the Ordinary Kriging (OK) interpolation method. Variogram parameters for each metal were used in each of these passes and aligned to the domain wireframe. Inverse Distance square (ID^2) and Nearest Neighbour (NN) interpolations were also completed for validation purposes.

To reduce some of the negative kriging weights, the data was subsequently de-clustered using an octant search for Pass 1 and Pass 2. The number of composites used for the interpolation was also adjusted for these passes.

Table 14-27 shows estimation parameters for each pass used to estimate metal grades.

Table 14-27: Interpolation Strategy for Dynamic Anisotropy– Z87 Zone.

	1st Running	2nd Running	3rd Running
Minimum number of samples	6	3	2
maximum number of samples	18	18	18
maximum number of samples per Drillhole	3	2	2
Maximum average distance of samples	60	120	120
Percent Partition of Original Search Ellipse Size (Au, Ag, and Cu)	60%	80%	100%
Ag (g/t) - Z87 Domain 1 (m)	78	104	130
Au (g/t) - Z87 Domain 1 (m)	70	93	116
Cu (%) - Z87 Domain 1 (m)	88	117	146
Ag (g/t) - Z87 Domain 2 (m)	81	117	146
Au (g/t) - Z87 Domain 2 (m)	49	65	81
Cu (%) - Z87 Domain 2 (m)	60	80	100
Ag (g/t) - Z87 Domain 3 (m)	116	154	193
Au (g/t) - Z87 Domain 3 (m)	57	154	193
Cu (%) - Z87 Domain 3 (m)	107	143	179
Ag (g/t) - Z87 Domain 4 (m)	99	132	165
Au (g/t) - Z87 Domain 4 (m)	91	122	152
Cu (g/t) - Z87 Domain 4 (m)	59	122	152
Ag (g/t) - Z87 Domain 5 (m)	74	98	123
Au (g/t) - Z87 Domain 5 (m)	78	104	130
Cu (%) - Z87 Domain 5 (m)	84	112	140
Ag (g/t) - Z87 Domain 6 (m)	132	112	140
Au (g/t) - Z87 Domain 6 (m)	130	173	216
Cu (%) - Z87 Domain 6 (m)	61	81	101

Table 14-28 to Table 14-30 present the search ellipses used to estimate gold by mineralized domain.

Table 14-28: Search Ellipse Parameters for Gold by Mineralized Domain – Z87 Zone

Zone	Bearing	Plunge	Dip	Major/Semi	Major/Minor
LG Halo	119	74	-25	1.75	3.75
v11	100	15	0	1.75	3.75
v11 HG-1	100	15	0	1.75	3.75
v11 HG-2	100	15	0	1.75	3.75
v13	100	15	0	1.75	3.75
v13 HG	100	15	0	1.75	3.75
v14	100	65	0	2.00	3.22
v14 HG	100	65	0	2.00	3.22
v17	100	65	0	2.00	3.22
v18	100	65	0	2.00	3.22
v19	100	65	0	2.00	3.22
v20	100	15	0	1.75	3.75
v21	100	15	0	1.75	3.75
v22	100	65	0	2.00	3.22
v24	100	65	0	2.00	3.22
v25	95	-80	10	1.00	8.2973
v26	100	65	0	2.00	3.2173
v27	90	80	0	1.98	3.2851
v28	90	80	0	1.98	3.2851
v29	95	-80	10	1.00	8.2973
v30	0	0	-80	2.12	3.2177
v31	0	0	-80	2.12	3.2177

Table 14-29: Search Ellipse Parameters for Copper by Mineralized Domain – Z87 Zone

Zone	Bearing	Plunge	Dip	Major/Semi	Major/Minor
LG Halo	100	75	-10	1.79	2.71
v11	260	80	0	1.79	2.71
v11 HG-1	260	80	0	1.79	2.71
v11 HG-2	260	80	0	1.79	2.71
v13	260	80	0	1.79	2.71
v13 HG	260	80	0	1.79	2.71
v20	260	80	0	1.79	2.71
v21	260	80	0	1.79	2.71
v14	100	60	0	2.03	4.35
v14 HG	100	60	0	2.03	4.35
v17	100	60	0	2.03	4.35
v18	100	60	0	2.03	4.35
v19	100	60	0	2.03	4.35
v22	100	60	0	2.03	4.35
v24	100	60	0	2.03	4.35
v26	100	60	0	2.03	4.35
v27	100	80	0	1.29	5.34
v28	100	80	0	1.29	5.34
v25	95	75	0	1.42	8.62
v29	95	75	0	1.42	8.62
v30	100	70	-10	1.80	4.06
v31	100	70	-10	1.80	4.06

Table 14-30: Search Ellipse Parameters for Silver by Mineralized Domain – Z87 Zone

Zone	Bearing	Plunge	Dip	Major/Semi	Major/Minor
LG Halo	127	79	-15	1.40	2.76
v11	105	70	-15	1.40	2.76
v11HG-1	105	70	-15	1.40	2.76
v11HG-2	105	70	-15	1.40	2.76
v13	105	70	-15	1.40	2.76
v13HG	105	70	-15	1.40	2.76
v20	105	70	-15	1.40	2.76
v21	105	70	-15	1.40	2.76
v14	105	65	0	2.00	5.56
v14HG	105	65	0	2.00	5.56
v17	105	65	0	2.00	5.56
v18	105	65	0	2.00	5.56
v19	105	65	0	2.00	5.56
v22	105	65	0	2.00	5.56
v24	105	65	0	2.00	5.56
v26	105	65	0	2.00	5.56
v27	100	-75	0	2.27	2.86
v28	100	-75	0	2.27	2.86
v25	100	80	0	1.96	3.46
v29	100	80	0	1.96	3.46
v30	15	0	-80	1.59	4.14
v31	15	0	-80	1.59	4.14

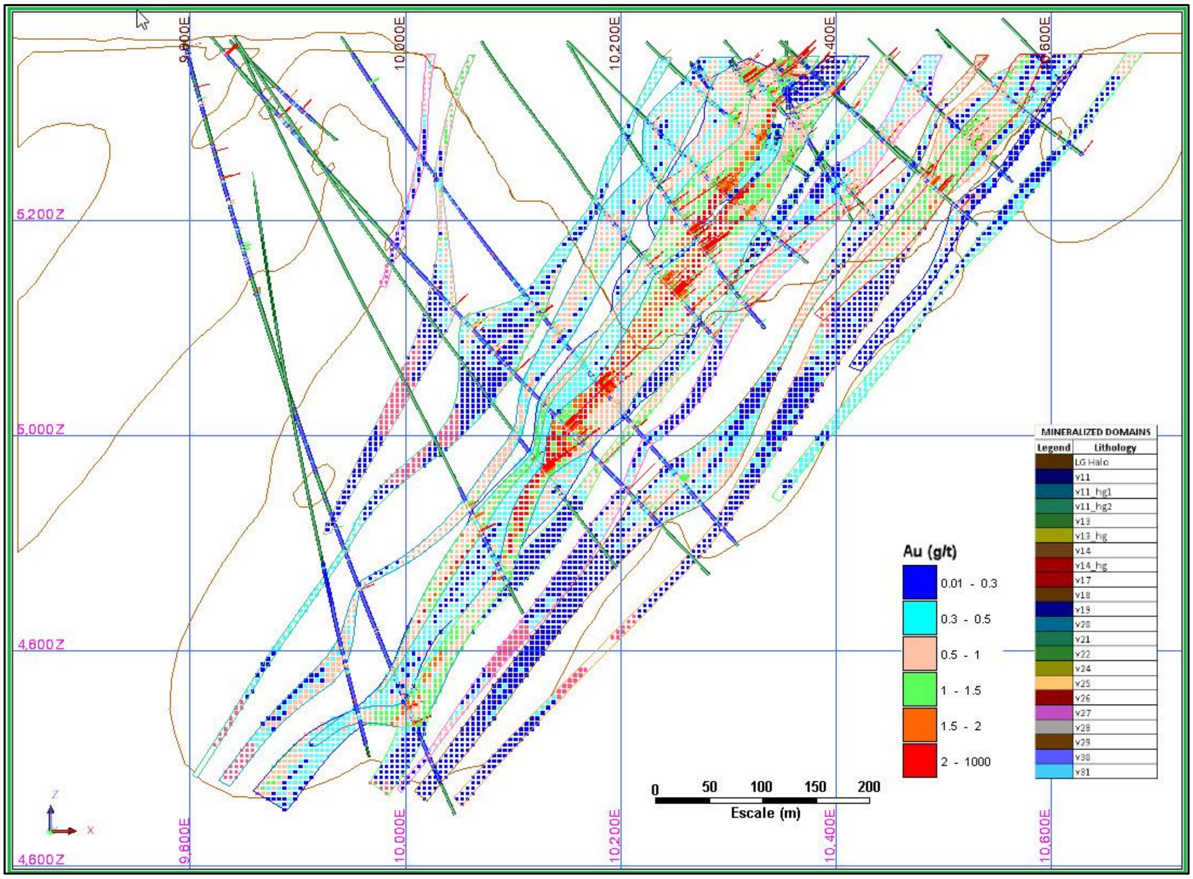
Block Model Validation

The Z87 Zone grade models were validated by the following methods:

- visual comparison of colour-coded block model grades with composite grades on sections and plans
- comparison of the global mean block grades for OK, ID^2, NN models, composite, and raw assay grades
- comparison using swath plots to investigate local bias in the estimate

The visual comparison of block model grades on sections and plans indicated a good correlation between drill hole grades and resource model grades (Figure 14-7).

Figure 14-7: Cross-Section (13400 mN) – Z87 Zone



Source: AGP (2023)

Table 14-31 shows the mean grade statistics for the composite values, NN, ID^2, and OK models. Statistics for the gold and copper composite mean grades compare well to the raw assay grades, with a normal reduction in values due to smoothing, related to volume variance. The block model mean grade, when compared against the composites, showed a normal reduction in values. More importantly, the grade of the NN, ID^2, and OK models are less than 1% of each other, indicating the methodology used did not introduce a local bias into the estimate.

Table 14-31: Global Comparison of Mean Grades in Complete Model – Z87 Zone.

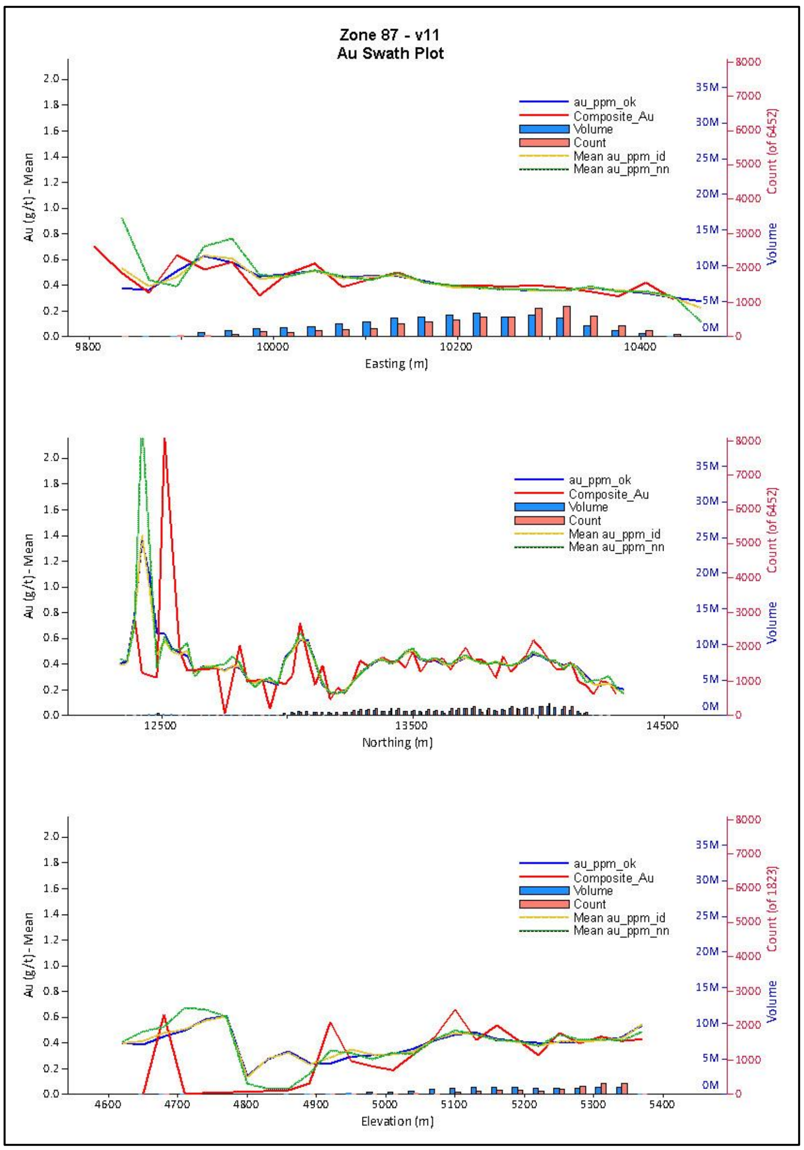
Rocktype	Au (g/t) Composite	Au (g/t) Ok	Au (g/t) ID	Au (g/t) NN	Cu (%) Composite	Cu (%) Ok	Cu (%) ID	Cu (%) NN	Ag (g/t) Composite	Ag (g/t) Ok	Ag (g/t) ID	Ag (g/t) NN
LG Halo	0.111	0.098	0.098	0.097	0.017	0.018	0.018	0.018	0.494	0.481	0.475	0.473
v11	0.406	0.396	0.396	0.409	0.054	0.054	0.055	0.056	0.295	0.892	0.893	0.914
v11 HG-1	1.386	1.311	1.305	1.306	0.168	0.141	0.140	0.142	0.881	1.488	1.469	1.483
v11 HG-2	0.984	0.844	0.849	0.852	0.085	0.088	0.089	0.087	1.708	1.443	1.413	1.440
v13	0.491	0.445	0.442	0.439	0.062	0.057	0.057	0.057	1.014	0.743	0.748	0.751
v13 HG	1.095	0.792	0.778	0.791	0.100	0.075	0.075	0.076	0.954	1.362	1.352	1.341
v14	0.428	0.379	0.381	0.388	0.044	0.048	0.048	0.050	1.288	1.035	1.035	1.071
v14 HG	1.031	0.774	0.801	0.829	0.056	0.045	0.044	0.043	1.240	1.113	1.097	1.058
v17	0.493	0.429	0.432	0.464	0.057	0.046	0.046	0.047	1.919	0.951	0.988	1.175
v18	0.426	0.343	0.344	0.334	0.029	0.028	0.028	0.028	1.012	0.663	0.662	0.677
v19	0.357	0.370	0.364	0.387	0.022	0.020	0.019	0.020	0.758	1.057	1.034	1.041
v20	0.392	0.342	0.340	0.341	0.039	0.047	0.047	0.046	0.995	1.107	1.092	1.077
v21	0.263	0.230	0.228	0.237	0.031	0.037	0.037	0.042	0.875	0.673	0.684	0.755
v22	0.444	0.430	0.454	0.475	0.021	0.021	0.021	0.023	0.588	0.773	0.797	0.897
v24	0.381	0.392	0.377	0.484	0.035	0.037	0.036	0.037	0.833	1.947	1.912	2.041
v25	0.243	0.288	0.283	0.280	0.061	0.077	0.076	0.076	1.772	0.795	0.784	0.813
v26	0.289	0.304	0.316	0.316	0.056	0.057	0.058	0.058	0.653	1.603	1.639	1.660
v27	0.444	0.451	0.451	0.457	0.018	0.017	0.016	0.017	1.708	0.694	0.734	0.842
v28	0.271	0.257	0.254	0.276	0.018	0.020	0.020	0.022	0.834	0.507	0.498	0.545
v29	0.171	0.208	0.203	0.225	0.037	0.047	0.045	0.049	0.470	0.728	0.723	0.776
v30	0.361	0.416	0.426	0.408	0.098	0.100	0.100	0.100	0.613	1.481	1.445	1.480
v31	0.332	0.333	0.326	0.335	0.040	0.038	0.038	0.039	1.405	0.795	0.795	0.803

Swath plots were used as comparison of grade profiles of the composite values, and estimated grades allow for a visual verification of an over or under estimation of the block grades at the global and local scales. A qualitative assessment of the smoothing and variability of the estimates can also be observed from the plots.

The swath plots show good agreement with the three interpolation methodologies showing no major local bias. The peaks and valleys on the block grades and composite values are well correlated.

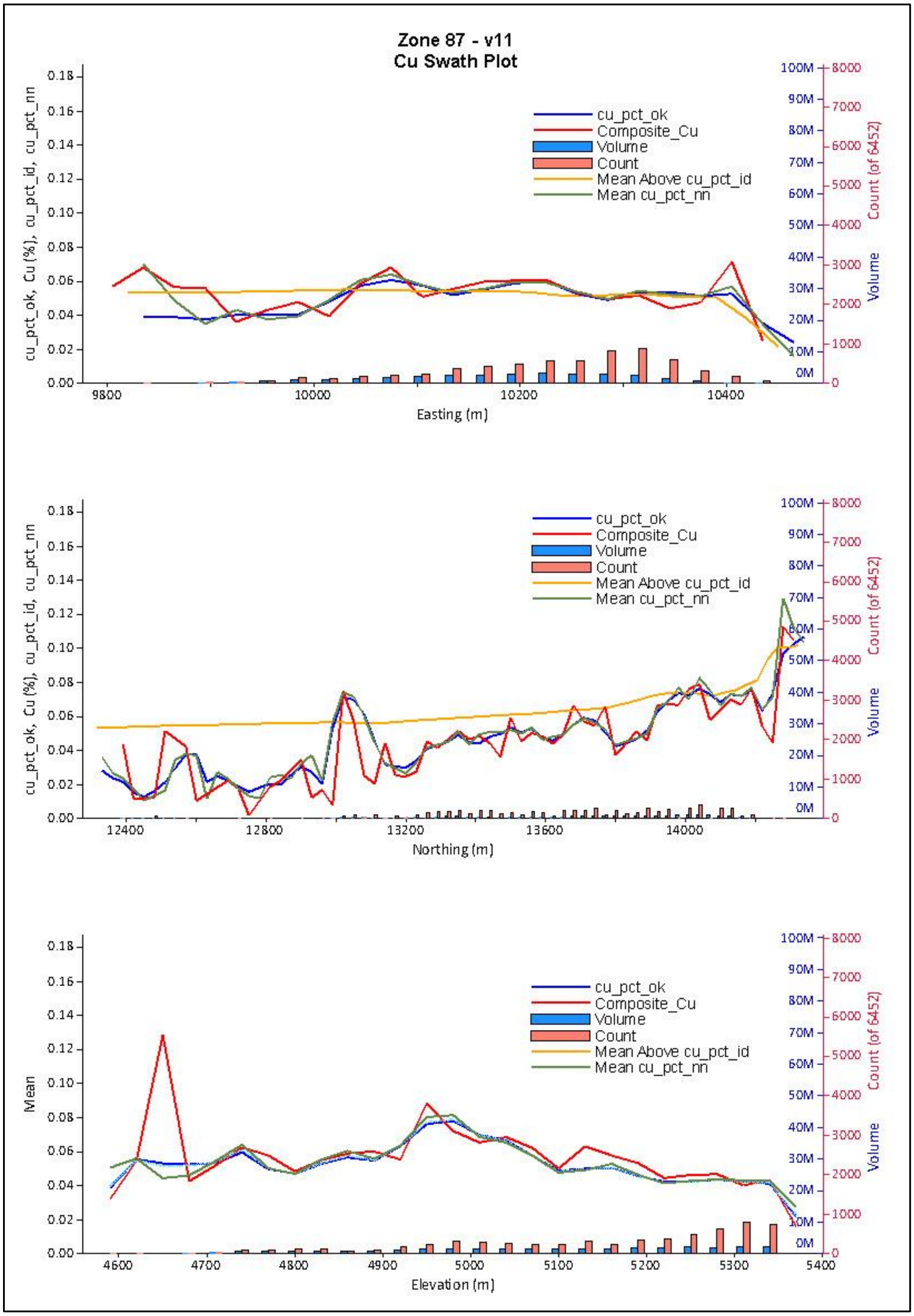
Figure 14-8 and Figure 14-9 present the swath plots for gold and copper, respectively, for domain v11. Figure 14-10 and Figure 14-11 present the swath plots for gold and copper, respectively, for domain v17.

Figure 14-8: Swath Plot for Gold, by Easting, Northing and Elevation; Domain v11 – Z87 Zone



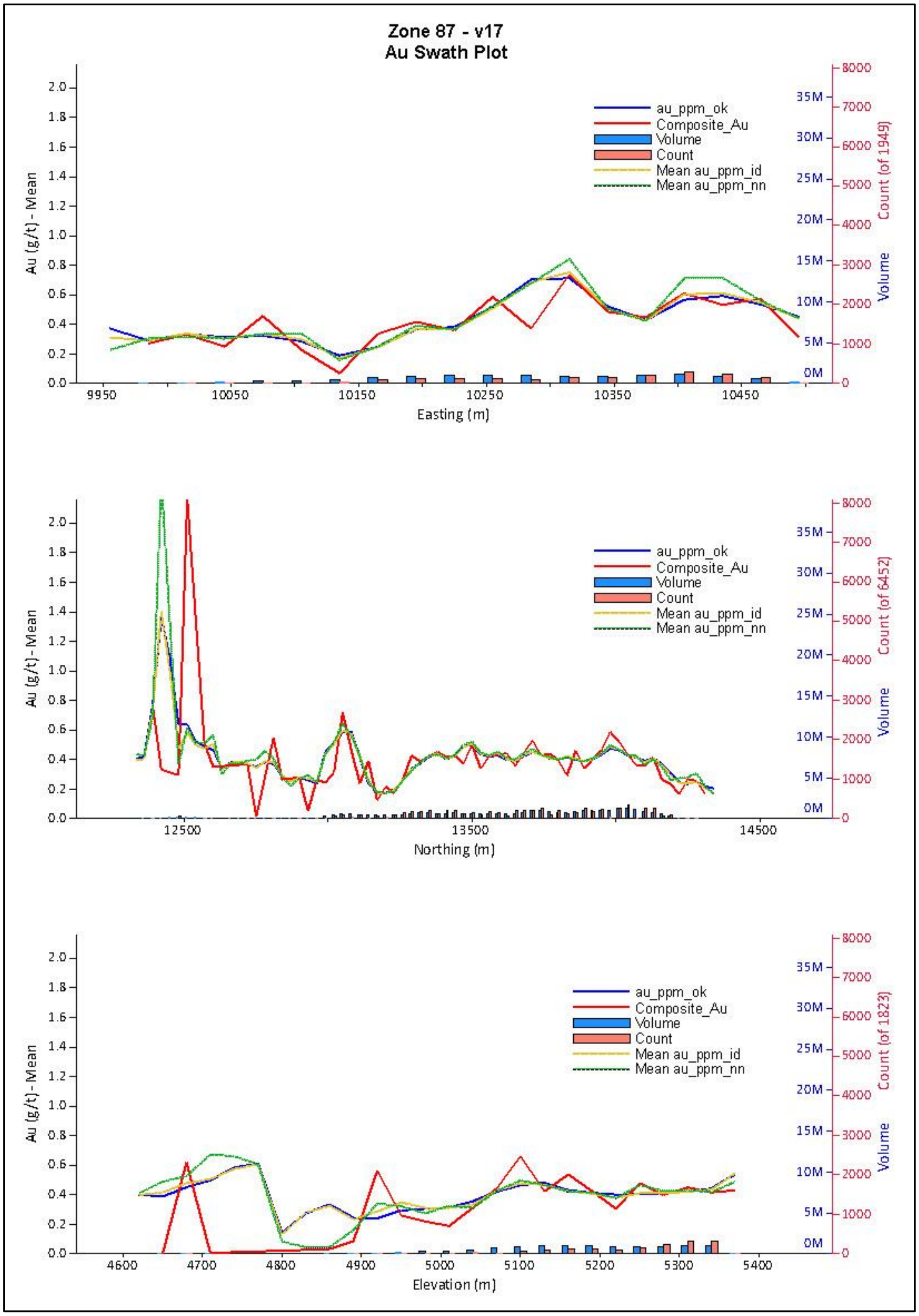
Source: AGP (2023)

Figure 14-9: Swath Plot for Copper, by Easting, Northing and Elevation; Domain v11 – Z87 Zone



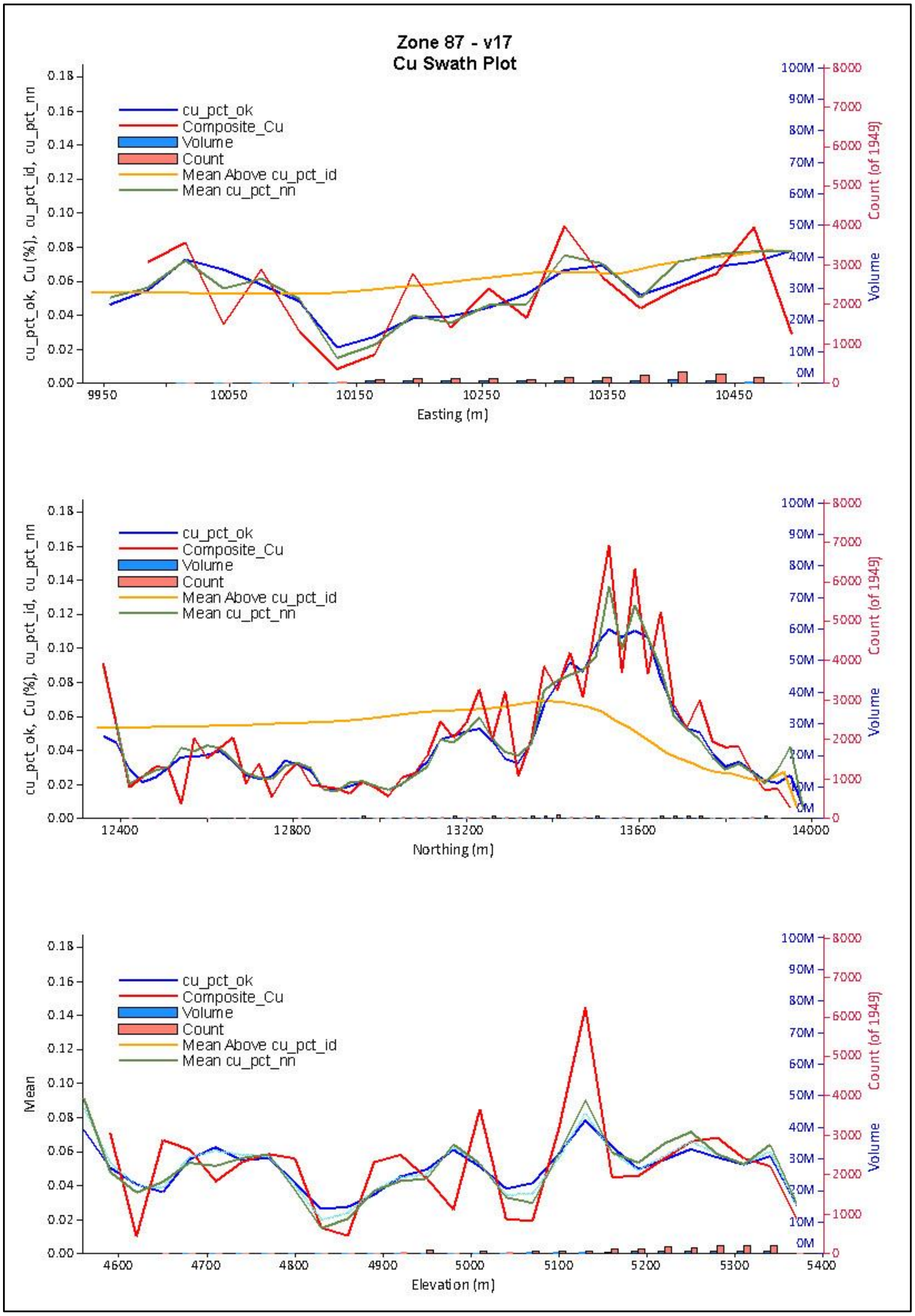
Source: AGP (2023)

Figure 14-10: Swath Plot for Gold, by Easting, Northing and Elevation; Domain v17 – Z87 Zone



Source: AGP (2023)

Figure 14-11: Swath Plot for Copper, by Easting, Northing and Elevation; Domain v17 – Z87 Zone



Source: AGP (2023)

14.6 J Zone

14.6.1 Interpretation

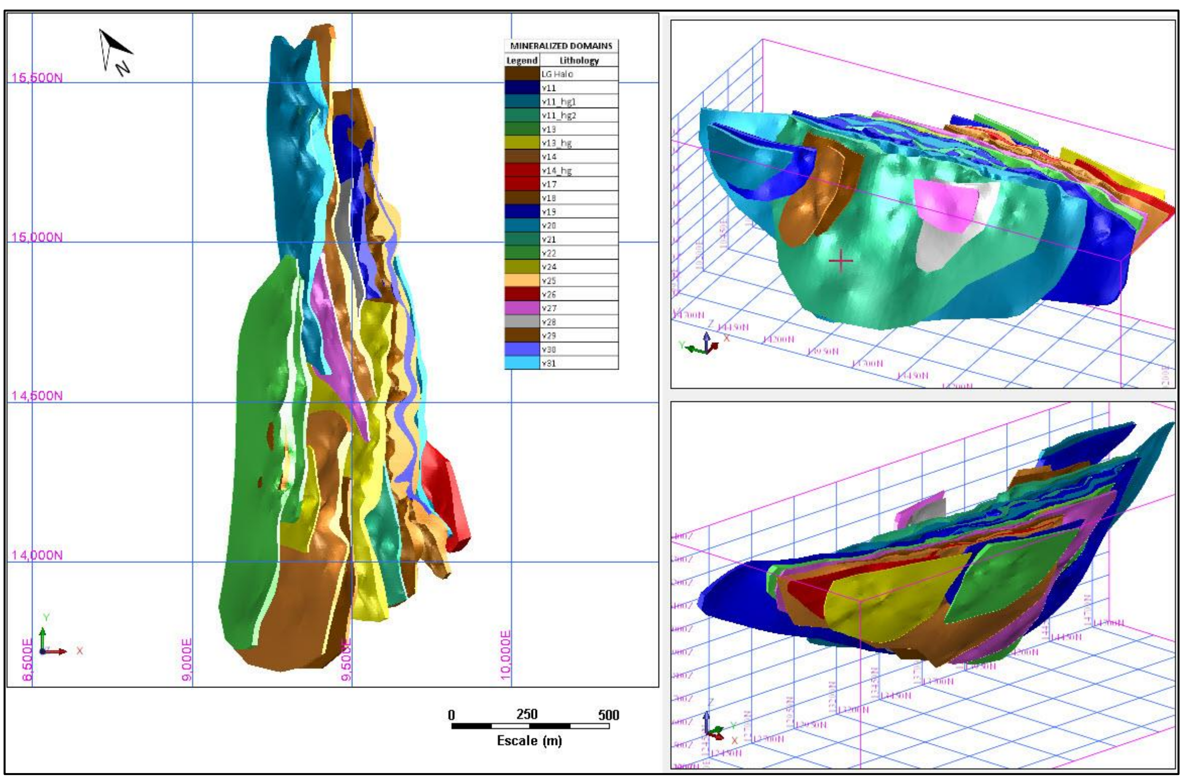
The mineralized domains at J Zone were interpreted by Troilus personnel. The interpreted wireframes were completed using Leapfrog Geo where grades were captured using a minimum grade of 0.3 g/t AuEQ with a minimum thickness of 5 m. Several higher-grade domains were captured using a minimum grade of 1.0 g/t AuEQ. The AuEQ formula used for the gold equivalent formula for the J Zone from the PEA Report (AGP, 2020a) as follows:

$$AuEQ = Au\ grade + (1.2979 * Cu\ grade) + (0.0108 * Ag\ grade)$$

All mineralized domain envelopes were created above pre-mining topography and clipped to the overburden bottom surface and then, to the pit topography. A total of 20 3D wireframes describe the mineralized domains in the J Zone. This includes a surrounding low-grade domain (or halo), created based on grades greater than 0.07 g/t Au. The mineralization is disseminated and shows enrichment in what could be described as mineralized corridors without sharp boundaries. AGP considers the wireframes suitable to estimate resources.

Figure 14-12 shows the mineralized domain wireframes for the J Zone. Table 14-32 summarizes the mineralized domains and domain codes for the J Zone.

Figure 14-12: Mineralized Domains – J Zone



Source: AGP (2023)

Table 14-32: Mineralized Zones and Domain Codes – J Zone

Mineralized Zone	Domain Code	Mineralized Zone	Domain Code
LG Halo	2000	v07	2070
v01	2010	v08	2080
v01 HG	2015	v10	2100
v02	2020	v11	2110
v03	2030	v12	2120
v04	2040	v13	2130
v04 HG	2045	v15	2150
v05	2050	v16	2160
v05 HG	2055	v17	2170
v06	2060	v18	2180

14.6.2 Exploratory Data Analysis

Raw Assays

The drill hole database for the mineralized domains in the J Zone, consists of 45,836 raw assay values for each metal: gold, copper, and silver. The assay values reported below detection limit were assigned half the detection limit for statistical analysis and grade estimation. Any missing values were assigned half the detection limit.

Table 14-33 to Table 14-35 presents the descriptive statistics for gold, copper, and silver, respectively, by mineralized domain in J Zone.

Table 14-33: Descriptive Statistics for Gold assays (g/t) by the Mineralized Domains – J Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
LG Halo	16,369	0	61.98	0.150	0.060	0.84	5.61
v01	8,411	0	46.06	0.494	0.270	1.26	2.54
v01 HG	2,610	0	69.88	1.284	0.701	2.65	2.06
v02	3,101	0	21.66	0.296	0.176	0.63	2.13
v03	1,830	0	15.40	0.340	0.180	0.69	2.03
v04	2,736	0	22.40	0.359	0.215	0.76	2.12
v04 HG	135	0.016	20.84	1.303	0.671	2.24	1.72
v05	847	0	5.05	0.375	0.205	0.55	1.48
v05 HG	229	0	2.86	0.580	0.468	0.54	0.93
v06	1,095	0	94.13	0.557	0.180	3.25	5.84
v07	752	0.001	5.88	0.290	0.189	0.41	1.40
v08	2,908	0	21.60	0.570	0.293	1.11	1.94
v10	855	0	8.03	0.430	0.185	0.77	1.80
v11	641	0	7.26	0.372	0.180	0.61	1.65
v12	329	0	1.385	0.206	0.158	0.19	0.90
v13	1,039	0	33.30	0.300	0.174	1.13	3.76
v15	656	0	11.80	0.538	0.196	1.26	2.34
v16	441	0	109.01	0.586	0.140	5.27	8.99
v17	401	0.001	3.78	0.185	0.099	0.35	1.88
v18	451	0	4.66	0.205	0.090	0.42	2.02

Table 14-34: Descriptive Statistics for Copper assays (%) by the Mineralized Domains – J Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
LG Halo	16,369	0	2.12	0.03	0.02	0.05	1.81
v01	8,411	0	2.01	0.05	0.03	0.06	1.27
v01 HG	2,610	0	1.60	0.06	0.04	0.08	1.27
v02	3,101	0	1.18	0.07	0.05	0.07	0.99
v03	1,830	0	0.37	0.05	0.04	0.04	0.86
v04	2,736	0	0.64	0.06	0.04	0.06	1.06
v04 HG	135	0.001	0.43	0.06	0.04	0.06	1.01
v05	847	0	0.77	0.06	0.02	0.09	1.55
v05 HG	229	0	0.97	0.19	0.16	0.19	1.00
v06	1,095	0.001	0.77	0.06	0.04	0.08	1.25
v07	752	0	0.66	0.07	0.05	0.07	1.09
v08	2,908	0	1.12	0.06	0.03	0.08	1.38
v10	855	0	0.99	0.07	0.04	0.10	1.38
v11	641	0	1.02	0.06	0.03	0.09	1.53
v12	329	0	0.78	0.09	0.07	0.10	1.03
v13	1,039	0	1.09	0.08	0.06	0.09	1.15
v15	656	0	0.90	0.07	0.04	0.11	1.49
v16	441	0	1.23	0.05	0.04	0.07	1.42
v17	401	0.001	0.24	0.04	0.03	0.03	0.80
v18	451	0	0.78	0.04	0.03	0.06	1.38

Table 14-35: Descriptive Statistics for Silver assays (g/t) by the Mineralized Domains – J Zone.

Domain	Count	Min	Max	Mean	Median	StDev	CV
LG Halo	16,369	0	720.00	0.52	0.25	5.73	11.07
v01	8,411	0	206.00	0.83	0.50	2.50	3.03
v01 HG	2,610	0	24.20	1.07	0.80	1.13	1.05
v02	3,101	0	46.90	0.89	0.60	1.43	1.61
v03	1,830	0	15.40	0.66	0.30	0.90	1.37
v04	2,736	0	31.80	0.78	0.50	1.03	1.33
v04 HG	135	0	7.20	0.99	0.78	0.98	0.99
v05	847	0	15.70	1.09	0.50	1.64	1.51
v05 HG	229	0	17.30	2.83	2.15	2.90	1.03
v06	1,095	0	10.20	0.76	0.40	1.03	1.35
v07	752	0	6.30	0.56	0.25	0.60	1.08
v08	2,908	0	25.60	1.01	0.60	1.45	1.43
v10	855	0	13.90	1.12	0.60	1.66	1.48
v11	641	0	14.50	0.88	0.25	1.42	1.60
v12	329	0	7.90	0.68	0.25	0.88	1.29
v13	1,039	0	8.60	0.62	0.25	0.76	1.22
v15	656	0	18.40	1.36	0.80	1.79	1.31
v16	441	0	42.80	0.58	0.25	2.25	3.86
v17	401	0	535.00	1.80	0.25	26.71	14.81
v18	451	0	62.70	0.56	0.25	3.02	5.42

Capping Analysis

Capping analysis was carried out on each mineralized domain for gold, copper and silver by disintegration analysis, histogram, and probability plots. Capping was applied first to the raw data in several mineralized domains where necessary.

Table 14-36 presents the capping levels applied to the mineralized domains.

Table 14-36: Capping Levels by Domain – J Zone

Domain	Au (g/t)	Loss (%)	Cu (%)	Loss (%)	Ag (g/t)	Loss (%)
LG Halo	2.30 (86)	15.0	0.83 (5)	6.8	12.50 (13)	11.0
v01	10.90 (13)	4.1	0.90 (2)	0.4	15.80 (6)	3.5
v01 HG	10.90 (23)	7.7	0.90 (3)	0.9		
v02	10.90 (2)	1.4	0.90 (1)	0.3		
v03	10.90 (1)	0.7				
v04					15.8 (1)	0.8
v04 HG						
v05						
v05 HG						
v06	5.84 (10)	28				
v07						
v08			0.90 (2)	0.3		
v10						
v11						
v12						
v13	10.90 (1)	7.2	0.90 (2)	0.4		
v15	6.49 (8)	3.7	0.90 (1)	0.1		
v16	4.50 (4)	48.0			5.40 (2)	19.0
v17					6.30 (2)	7.4
v18					7.8 (2)	29.0

Table 14-37 to Table 14-39 present the descriptive statistics for gold, copper, and silver by mineralized domains, respectively.

Table 14-37: Descriptive Statistics for Capped raw Gold Assays (g/t Au) by Mineralized Domain – J Zone

Mineralized Zone	Count	Min	Max	Mean	Median	Variance	StDev	CV
LG Halo	16,369	0	2.30	0.13	0.06	0.06	0.25	1.91
v01	8,411	0	10.90	0.47	0.27	0.65	0.81	1.70
v01 HG	2,610	0	10.90	1.20	0.72	2.58	1.61	1.33
v02	3,101	0	10.90	0.29	0.18	0.28	0.53	1.80
v03	1,830	0	10.90	0.34	0.18	0.42	0.65	1.90
v04	2,736	0	22.40	0.36	0.22	0.58	0.76	2.12
v04 HG	135	0.02	20.84	1.39	0.74	5.43	2.33	1.67
v05	847	0	5.05	0.37	0.20	0.30	0.55	1.48
v05 HG	229	0	2.86	0.59	0.47	0.30	0.55	0.93
v06	1,095	0	5.84	0.40	0.18	0.59	0.77	1.93
v07	752	0	5.88	0.29	0.19	0.17	0.41	1.40
v08	2,908	0	21.60	0.57	0.29	1.23	1.11	1.96
v10	855	0	7.29	0.42	0.19	0.53	0.73	1.73
v11	641	0	7.26	0.37	0.18	0.38	0.62	1.65
v12	329	0	1.39	0.21	0.16	0.03	0.19	0.90
v13	1,039	0	10.90	0.28	0.17	0.33	0.58	2.07
v15	656	0	10.90	0.55	0.20	1.60	1.27	2.31
v16	441	0	10.90	0.35	0.14	1.08	1.04	2.94
v17	401	0	3.78	0.18	0.10	0.12	0.35	1.89
v18	451	0	4.66	0.20	0.09	0.17	0.42	2.03

Table 14-38: Descriptive Statistics for Capped raw Copper Assays (%Cu) by Mineralized Domain – J Zone

Mineralized Zone	Count	Min	Max	Mean	Median	Variance	StDev	CV
LG Halo	16,369	0	0.83	0.03	0.02	-	0.04	1.55
v01	8,411	0	0.90	0.05	0.03	-	0.05	1.18
v01 HG	2,610	0	0.90	0.06	0.04	-	0.07	1.15
v02	3,101	0	0.90	0.07	0.06	-	0.07	0.97
v03	1,830	0	0.37	0.05	0.04	-	0.04	0.86
v04	2,736	0	0.64	0.06	0.04	-	0.06	1.06
v04 HG	135	0	0.43	0.06	0.04	-	0.06	1.01
v05	847	0	0.77	0.06	0.02	0.01	0.09	1.55
v05 HG	229	0	0.97	0.19	0.16	0.04	0.19	0.99
v06	1,095	0	0.77	0.06	0.04	0.01	0.08	1.26
v07	752	0	0.66	0.07	0.05	0.01	0.07	1.10
v08	2,908	0	0.90	0.05	0.03	0.01	0.07	1.36
v10	855	0	0.99	0.07	0.04	0.01	0.10	1.38
v11	641	0	1.02	0.06	0.03	0.01	0.09	1.54
v12	329	0	0.78	0.09	0.07	0.01	0.10	1.02
v13	1,039	0	0.90	0.08	0.06	0.01	0.08	1.11
v15	656	0	0.90	0.07	0.04	0.01	0.11	1.48
v16	441	0	1.23	0.05	0.04	-	0.07	1.42
v17	401	0	0.24	0.04	0.03	-	0.03	0.80
v18	451	0	0.56	0.04	0.03	-	0.05	1.19

Table 14-39: Descriptive Statistics for Capped raw Silver Assays (g/t Ag) by Mineralized Domain – J Zone

Mineralized Zone	Count	Min	Max	Mean	Median	Variance	StDev	CV
LG Halo	16,369	0	12.50	0.39	0.25	0.37	0.61	1.57
v01	8,411	0	15.80	0.80	0.50	0.92	0.96	1.21
v01 HG	2,610	0	24.20	1.08	0.80	1.29	1.13	1.05
v02	3,101	0	46.90	0.90	0.60	2.08	1.44	1.60
v03	1,830	0	15.40	0.66	0.32	0.81	0.90	1.37
v04	2,736	0	15.80	0.77	0.50	0.80	0.89	1.16
v04 HG	135	0	7.20	1.00	0.77	1.03	1.01	1.01
v05	847	0	15.70	1.09	0.50	2.70	1.64	1.51
v05 HG	229	0	17.30	2.85	2.20	8.48	2.91	1.02
v06	1,095	0	10.20	0.77	0.40	1.07	1.03	1.35
v07	752	0	6.30	0.56	0.25	0.37	0.61	1.09
v08	2,908	0	25.60	1.00	0.60	2.11	1.45	1.45
v10	855	0	13.90	1.13	0.60	2.78	1.67	1.48
v11	641	0	14.50	0.88	0.25	2.01	1.42	1.60
v12	329	0	7.90	0.68	0.25	0.76	0.87	1.28
v13	1,039	0	8.60	0.62	0.25	0.58	0.76	1.22
v15	656	0	18.40	1.39	0.80	3.27	1.81	1.30
v16	441	0	5.40	0.47	0.25	0.43	0.66	1.40
v17	401	0	6.30	0.47	0.25	0.54	0.73	1.56
v18	451	0	7.80	0.41	0.25	0.40	0.64	1.54

Composites

Composites were created within the mineralized domains, starting from the domain intersection. Composite lengths were adjusted equally across the domain intersection.

Table 14-40 to Table 14-42 show the descriptive statistics by domains for the 2 m capped composite values for the J Zone

Table 14-40: Descriptive Statistics for Capped Au (g/t) Composite Values by Mineralized Domain – J Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
LG Halo	30,615	0	2.30	0.11	0.06	0.19	1.64
v01	5,439	0	10.90	0.47	0.31	0.64	1.36
v01 HG	1,528	0.003	10.90	1.21	0.85	1.31	1.08
v02	2,040	0	10.90	0.31	0.20	0.48	1.57
v03	1,127	0	10.90	0.36	0.21	0.63	1.74
v04	1,529	0	13.08	0.34	0.23	0.56	1.63
v04 HG	71	0.046	20.84	1.60	0.87	2.68	1.68
v05	500	0	3.24	0.39	0.25	0.46	1.19
v05 HG	120	0	2.46	0.59	0.58	0.47	0.80
v06	637	0	5.84	0.40	0.22	0.62	1.58
v07	445	0.001	3.31	0.29	0.21	0.31	1.07
v08	1,910	0	10.23	0.55	0.33	0.79	1.43
v10	546	0	6.23	0.41	0.21	0.67	1.62
v11	389	0	3.92	0.33	0.170	0.48	1.43
v12	194	0.007	0.96	0.20	0.18	0.15	0.75
v13	631	0.003	6.598	0.30	0.20	0.51	1.73
v15	405	0.001	6.49	0.49	0.23	0.85	1.75
v16	303	0	3.93	0.28	0.16	0.45	1.57
v17	267	0.003	2.34	0.19	0.11	0.27	1.39
v18	277	0	2.48	0.18	0.09	0.29	1.61

Table 14-41: Descriptive Statistics for Capped Cu (%) Composite Values by Mineralized Domain – J Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
LG Halo	30,615	0	0.77	0.02	0.02	0.03	1.35
v01	5,439	0	0.90	0.05	0.04	0.05	1.00
v01 HG	1,528	0	0.90	0.06	0.05	0.06	1.02
v02	2,040	0	0.85	0.07	0.06	0.06	0.84
v03	1,127	0	0.34	0.05	0.04	0.04	0.79
v04	1,529	0	0.64	0.06	0.04	0.06	1.01
v04 HG	71	0.001	0.27	0.06	0.04	0.05	0.91
v05	500	0	0.64	0.06	0.03	0.09	1.38
v05 HG	120	0	0.86	0.19	0.17	0.17	0.90
v06	637	0.001	0.57	0.07	0.05	0.07	1.08
v07	445	0	0.47	0.07	0.05	0.06	0.89
v08	1,910	0	0.61	0.05	0.04	0.06	1.15
v10	546	0	0.56	0.07	0.04	0.09	1.25
v11	389	0	0.76	0.06	0.03	0.08	1.38
v12	194	0.003	0.51	0.09	0.07	0.08	0.82
v13	631	0.002	0.90	0.08	0.06	0.07	0.93
v15	405	0.000	0.83	0.07	0.04	0.10	1.33
v16	303	0	0.31	0.05	0.04	0.04	0.86
v17	267	0.001	0.17	0.04	0.03	0.03	0.68
v18	277	0	0.30	0.04	0.03	0.04	1.06

Table 14-42: Descriptive Statistics for Capped Ag (g/t) Composite Values by Mineralized Domain – J Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
LG Halo	30,615	0	11.71	0.36	0.25	0.48	1.33
v01	10,878	0	15.80	0.86	0.65	0.82	0.96
v01 HG	3,056	0	10.14	1.13	0.90	0.96	0.85
v02	2,040	0	23.85	0.93	0.70	1.07	1.16
v03	1,127	0	15.40	0.70	0.475	0.916	1.31
v04	1,529	0	10.10	0.77	0.525	0.755	0.98
v04 HG	71	0	4.05	1.06	1.01	0.84	0.79
v05	500	0	11.60	1.12	0.525	1.49	1.33
v05 HG	120	0	13.56	2.86	2.32	2.57	0.90
v06	637	0	8.85	0.82	0.546	0.95	1.17
v07	445	0	3.8	0.60	0.40	0.53	0.89
v08	1,910	0	13.95	1.05	0.725	1.26	1.20
v10	546	0	12.75	1.10	0.65	1.50	1.37
v11	389	0	10.4	0.83	0.25	1.17	1.42
v12	194	0.25	5.4	0.69	0.425	0.71	1.03
v13	631	0.10	7.3	0.64	0.50	0.60	0.94
v15	405	0.10	14.65	1.38	0.95	1.55	1.12
v16	303	0	5.20	0.48	0.25	0.59	1.22
v17	267	0	6.30	0.51	0.25	0.68	1.34
v18	277	0	4.60	0.38	0.25	0.48	1.24

Variography

Spatial analysis was performed on 2 m composites on J zone. A hard boundary was defined for zones v01, v04 and v05, separating their low-grade and high-grade domains. The mineralized domains were grouped in five different Grouped Domains. Table 14-43 presents the mineralized zoned within the Grouped Domains.

Table 14-43: Mineralized Zones within Grouped Domains – J Zone

Domain 1	Domain 2	Domain 4	Domain 5	Domain 6
v04	v02	v03	v01	LG Halo
v04_hg	v07	v06	v01_hg	
v05		v16	v08	
v05_hg		v17	v10	
v11			v15	
v12				
v13				
v18				

Experimental variograms were calculated for gold, copper, and silver and are oriented along the overall strike, dip, and across strike directions for each domain. Semi-automatic fitting was used for modelling the regionalized mineralization.

Tables 14-44 to Table 14-46 presents the variography parameters for gold, copper, and silver, by mineralized domain, respectively, in the J Zone.

Table 14-44: Variogram Parameters for Gold – J Zone

Zone	Anisotropy	Structure	Nugget	1st Sill	1st Range	2nd Sill	2nd Range	3rd Sill	3rd Range
LG Halo	Spherical	2	0.012456	0.006958	86.60	0.015935	165.96	-	-
v01	Spherical	2	0.350000	0.318047	58.89	0.087842	130.51	-	-
v01 HG	Spherical	2	0.350000	0.318047	58.89	0.087842	130.51	-	-
v02	Spherical	2	0.006625	0.005629	63.42	0.134663	127.49	-	-
v03	Spherical	2	0.153000	0.066194	26.50	0.142674	160.45	-	-
v04	Spherical	3	0.190000	0.076688	29.17	0.049375	108.41	0.080	138.434
v04 HG	Spherical	3	0.190000	0.076688	29.17	0.049375	108.41	0.080	138.434
v05	Spherical	3	0.190000	0.076688	29.17	0.049375	108.41	0.080	138.434
v05 HG	Spherical	3	0.190000	0.076688	29.17	0.049375	108.41	0.080	138.434
v06	Spherical	2	0.153000	0.066194	26.50	0.142674	160.45	-	-
v07	Spherical	2	0.006625	0.005629	63.42	0.134663	127.49	-	-
v08	Spherical	2	0.350000	0.318047	58.89	0.087842	130.51	-	-
v10	Spherical	2	0.350000	0.318047	58.89	0.087842	130.51	-	-
v11	Spherical	3	0.190000	0.076688	29.17	0.049375	108.41	0.080	138.434
v12	Spherical	3	0.190000	0.076688	29.17	0.049375	108.41	0.080	138.434
v13	Spherical	3	0.190000	0.076688	29.17	0.049375	108.41	0.080	138.434
v15	Spherical	2	0.350000	0.318047	58.89	0.087842	130.51	-	-
v16	Spherical	2	0.153000	0.066194	26.50	0.142674	160.45	-	-
v17	Spherical	2	0.153000	0.066194	26.50	0.142674	160.45	-	-
v18	Spherical	3	0.190000	0.076688	29.17	0.049375	108.41	0.080	138.434

Table 14-45: Variogram Parameters for Copper – J Zone

Zone	Anisotropy	Structure	Nugget	1st Sill	1st Range	2nd Sill	2nd Range	3rd Sill	3rd Range
LG Halo	Spherical	2	0.000251	0.000488	72.30	0.000334	157.31	-	-
v01	Spherical	2	0.001000	0.001500	38.90	0.000603	146.36	-	-
v01 HG	Spherical	2	0.001000	0.001500	38.90	0.000603	146.36	-	-
v02	Spherical	2	0.006625	0.005629	63.42	0.134663	127.49	-	-
v03	Spherical	3	0.000050	0.000592	46.79	0.001134	125.72	0.001	214.670
v04	Spherical	2	0.000800	0.003183	71.51	0.002348	237.60	-	-
v04 HG	Spherical	2	0.000800	0.003183	71.51	0.002348	237.60	-	-
v05	Spherical	2	0.000800	0.003183	71.51	0.002348	237.60	-	-
v05 HG	Spherical	2	0.000800	0.003183	71.51	0.002348	237.60	-	-
v06	Spherical	3	0.000050	0.000592	46.79	0.001134	125.72	0.001	214.670
v07	Spherical	2	0.006625	0.005629	63.42	0.134663	127.49	-	-
v08	Spherical	2	0.001000	0.001500	38.90	0.000603	146.36	-	-
v10	Spherical	2	0.001000	0.001500	38.90	0.000603	146.36	-	-
v11	Spherical	2	0.000800	0.003183	71.51	0.002348	237.60	-	-
v12	Spherical	2	0.000800	0.003183	71.51	0.002348	237.60	-	-
v13	Spherical	2	0.000800	0.003183	71.51	0.002348	237.60	-	-
v15	Spherical	2	0.001000	0.001500	38.90	0.000603	146.36	-	-
v16	Spherical	3	0.000050	0.000592	46.79	0.001134	125.72	0.001	214.670
v17	Spherical	3	0.000050	0.000592	46.79	0.001134	125.72	0.001	214.670
v18	Spherical	2	0.000800	0.003183	71.51	0.002348	237.60	-	-

Table 14-46: Variogram Parameters for Silver – J Zone

Zone	Anisotropy	Structure	Nugget	1st Sill	1st Range	2nd Sill	2nd Range	3rd Sill	3rd Range
LGHalo	Spherical	3	0.223322	0.060245	61.61	0.140225	126.06	0.1309	224.3680
v01	Spherical	2	0.150000	0.518475	55.34	0.592875	180.76	-	-
v01HG	Spherical	2	0.150000	0.518475	55.34	0.592875	180.76	-	-
v02	Spherical	2	0.080000	0.283249	43.92	0.811565	100.02	-	-
v03	Spherical	3	0.320000	0.202614	45.09	0.245839	112.55	0.264	157.449
v04	Spherical	3	0.500000	0.369516	36.19	0.018476	225.14	0.868	333.605
v04HG	Spherical	3	0.500000	0.369516	36.19	0.018476	225.14	0.868	333.605
v05	Spherical	3	0.500000	0.369516	36.19	0.018476	225.14	0.868	333.605
v05HG	Spherical	3	0.500000	0.369516	36.19	0.018476	225.14	0.868	333.605
v06	Spherical	3	0.320000	0.202614	45.09	0.245839	112.55	0.264	157.449
v07	Spherical	2	0.080000	0.283249	43.92	0.811565	100.02	-	-
v08	Spherical	2	0.150000	0.518475	55.34	0.592875	180.76	-	-
v10	Spherical	2	0.150000	0.518475	55.34	0.592875	180.76	-	-
v11	Spherical	3	0.500000	0.369516	36.19	0.018476	225.14	0.868	333.605
v12	Spherical	3	0.500000	0.369516	36.19	0.018476	225.14	0.868	333.605
v13	Spherical	3	0.500000	0.369516	36.19	0.018476	225.14	0.868	333.605
v15	Spherical	2	0.150000	0.518475	55.34	0.592875	180.76	-	-
v16	Spherical	3	0.320000	0.202614	45.09	0.245839	112.55	0.264	157.449
v17	Spherical	3	0.320000	0.202614	45.09	0.245839	112.55	0.264	157.449
v18	Spherical	3	0.500000	0.369516	36.19	0.018476	225.14	0.868	333.605

14.6.3 Block Model

Block Model Parameters

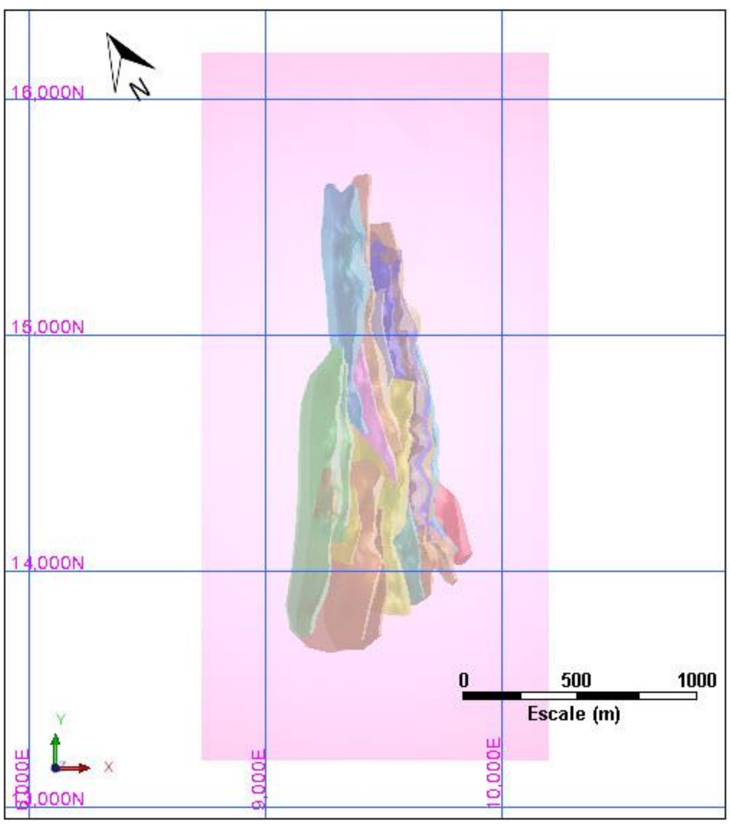
The block model for the J Zone deposit was set up with a block matrix of 5 m long by 5 m wide by 5 m high and was built using Geovia Surpac resource software. The block matrix was defined based on current drill hole spacing and on engineering considerations for an open pit operation and is considered suitable this purpose. The block model is in mine grid coordinates and is not rotated.

Table 14-47 summarizes the block model parameters. Figure 14-13 shows the block model extents over the J Zone.

Table 14-47: Block Model Parameters – J Zone

Type	Y	X	Z
Minimum Coordinates	13200 mN	8730 mE	4600 m
Maximum Coordinates	16200 mN	10200 mE	5500 m
Length (m)	3000	1470	900
Block Size (m)	5	5	5
Rotation	No rotation		
Number of Blocks	600	294	180

Figure 14-13: Block Model Extents – J Zone



Source: AGP (2023)

Estimation Parameters and Interpolation Strategy

The metal grades were interpolated in three passes using the 2 m capped composites. The metal grades were interpolated using ordinary kriging (OK) method. Variogram parameters for each metal was used in each of these passes and aligned to the domain wireframe. ID^2 and NN interpolation methods were also run for validation purposes.

Each pass required different minimum and maximum number of composites with a maximum of six composites per drill hole. Three drill holes were required to populate a block in pass 1 in order to guarantee spatial continuity. Table 14-48 summarizes the estimation parameters used for each pass to estimate the metal grades.

Table 14-48: Estimation Parameters for Gold, Silver, and Copper – J Zone

	1st Pass	2nd Pass	3rd Pass
Minimum number of samples	6	3	2
Maximum number of samples	18	18	18
Maximum number of samples per drill hole	3	2	2
Maximum average distance of samples	60	120	120
Percent Partition of Original Search Ellipse Size (Au, Ag and Cu)	60%	80%	100%
Ag (g/t) - Domain 1 (m)	200	267	334
Au (g/t) - Domain 1 (m)	83	110	138
Cu (%) - Domain 1 (m)	143	190	238
Ag (g/t) - Domain 2 (m)	60	190	238
Au (g/t) - Domain 2 (m)	76	102	127
Cu (%) - Domain 2 (m)	76	102	127
Ag (g/t) - Domain 3 (m)	200	267	334
Au (g/t) - Domain 3 (m)	83	267	334
Cu (%) - Domain 3 (m)	143	190	238
Ag (g/t) - Domain 4 (m)	94	126	157
Au (g/t) - Domain 4 (m)	96	128	160
Cu (%) - Domain 4 (m)	129	128	160
Ag (g/t) - Domain 5 (m)	109	145	181
Au (g/t) - Domain 5 (m)	79	105	131
Cu (%) - Domain 5 (m)	88	117	146
Ag (g/t) - LG Halo (m)	134	117	146
Au (g/t) - LG Halo (m)	100	133	166
Cu (%) - LG Halo (m)	94	126	157

Table 14-49 to Table 14-51 present the search ellipses used to estimate gold by mineralized domain.

Table 14-49: Search Ellipse Parameters for Gold by Mineralized Domain – J Zone

Zone	Bearing	Plunge	Dip	Major/Semi	Major/Minor
LG Halo	60	0	0	1.53	2.67
v01	65	82	10	1.53	2.67
v01 HG	65	82	10	1.53	2.67
v02	80	95	5	2.13	6.15
v03	-35	170	-80	2.13	6.15
v04	80	95	5	1.81	5.93
v04 HG	80	95	5	1.81	5.93
v05	80	95	5	1.81	5.93
v05 HG	80	95	5	1.81	5.93
v06	-35	170	-80	2.13	6.15
v07	80	95	5	2.13	6.15
v08	65	82	10	1.53	2.67
v10	65	82	10	1.53	2.67
v11	80	95	5	1.81	5.93
v12	80	95	5	1.81	5.93
v13	80	95	5	1.81	5.93
v15	65	82	10	1.53	2.67
v16	-35	170	-80	2.13	6.15
v17	-35	170	-80	2.13	6.15
v18	80	95	5	1.81	5.93

Table 14-50: Search Ellipse Parameters for Copper by Mineralized Domain – J Zone

Zone	Bearing	Plunge	Dip	Major/Semi	Major/Minor
LG Halo	60	100	0	1.05	3.81
v01	80	85	0	1.93	2.38
v01 HG	80	85	0	1.93	2.38
v02	80	95	5	2.13	6.15
v03	0	170	-80	2.91	5.81
v04	0	5	70	1.46	5.67
v04 HG	0	5	70	1.46	5.67
v05	0	5	70	1.46	5.67
v05 HG	0	5	70	1.46	5.67
v06	0	170	-80	2.91	5.81
v07	80	95	5	2.13	6.15
v08	80	85	0	1.93	2.38
v10	80	85	0	1.93	2.38
v11	0	5	70	1.46	5.67
v12	0	5	70	1.46	5.67
v13	0	5	70	1.46	5.67
v15	80	85	0	1.93	2.38
v16	0	170	-80	2.91	5.81
v17	0	170	-80	2.91	5.81
v18	0	5	70	1.46	5.67

Table 14-51: Search Ellipse Parameters for Silver by Mineralized Domain – J Zone

Zone	Bearing	Plunge	Dip	Major/Semi	Major/Minor
LG Halo	60	100	5	1.22	2.38
v01	65	85	0	1.22	2.38
v01 HG	65	85	0	1.22	2.38
v02	80	95	0	1.83	3.82
v03	0	160	-70	1.83	3.82
v04	-69	261	-15	1.54	3.29
v04 HG	-69	261	-15	1.54	3.29
v05	-69	261	-15	1.54	3.29
v05 HG	-69	261	-15	1.54	3.29
v06	0	160	-70	1.83	3.82
v07	80	95	0	1.83	3.82
v08	65	85	0	1.22	2.38
v10	65	85	0	1.22	2.38
v11	-69	261	-15	1.54	3.29
v12	-69	261	-15	1.54	3.29
v13	-69	261	-15	1.54	3.29
v15	65	85	0	1.22	2.38
v16	0	160	-70	1.83	3.82
v17	0	160	-70	1.83	3.82
v18	-69	261	-15	1.54	3.29

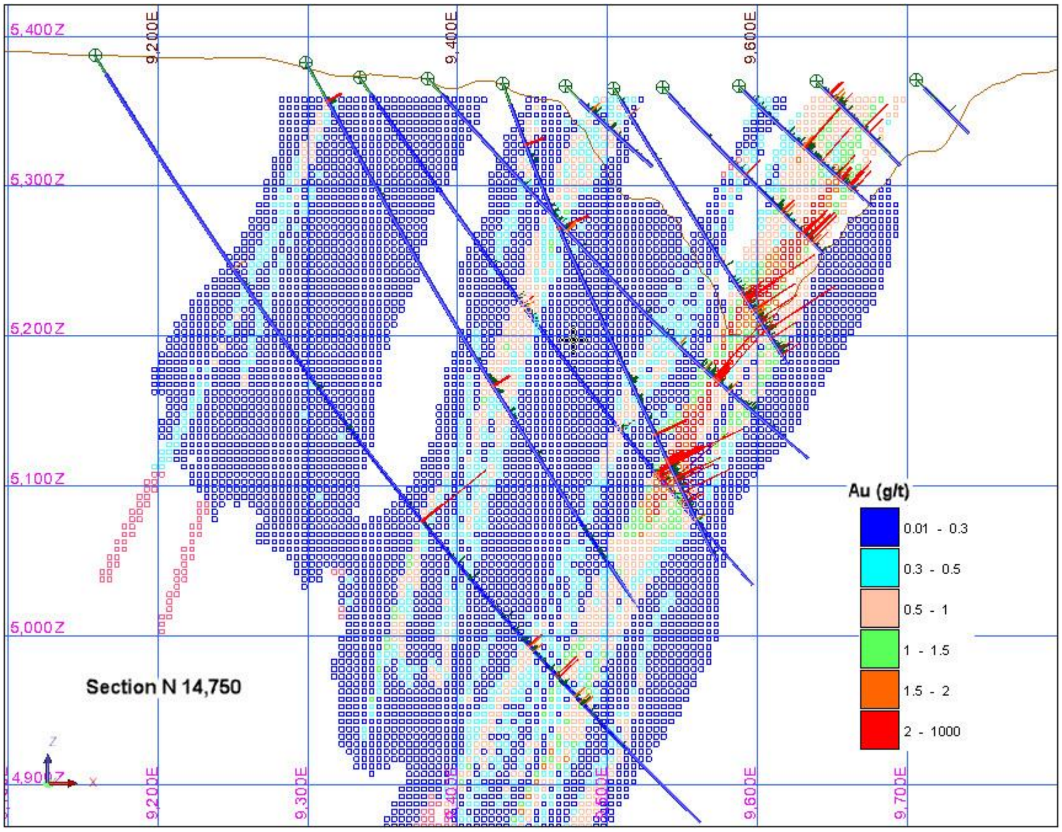
Block Model Validation

The J Zone grade models were validated by the following methods:

- visual comparison of colour-coded block model grades with composite grades on sections and plans
- comparison of the global mean block grades for OK, ID^2, NN models, composite, and raw assay grades
- comparison using swath plots to investigate local bias in the estimate

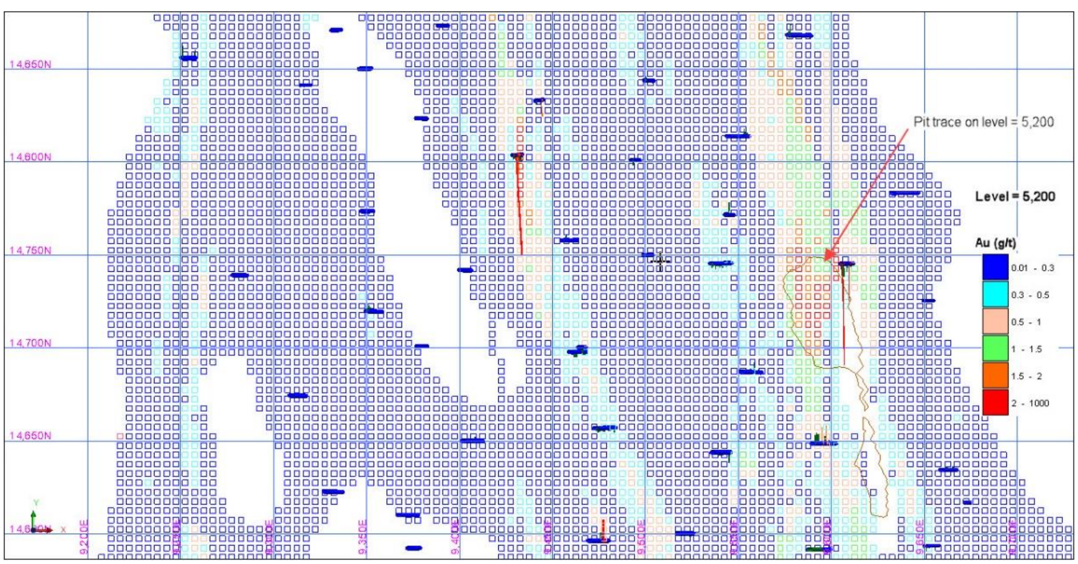
The block model grades, and the composites grades were visually inspected on plan and cross sections. Composite grades honour the block grades well within the high mineralized domains. Figure 14-14 and Figure 14-15 present a cross section (14750 mN) and plan view (5200 m), respectively.

Figure 14-14: Cross Section 14750 mN, showing gold grades – J Zone



Source: AGP (2023)

Figure 14-15: Plan Section 5200 m, showing gold grades – J Zone



Source: AGP (2023)

A series of validation tools were used in the block model validation, including statistical comparison of resource assay and block grade distributions, visual inspection, and comparison of block grades with assay grades and inspection of swath plots with block grades elevations and northings.

Mean Grade Comparison

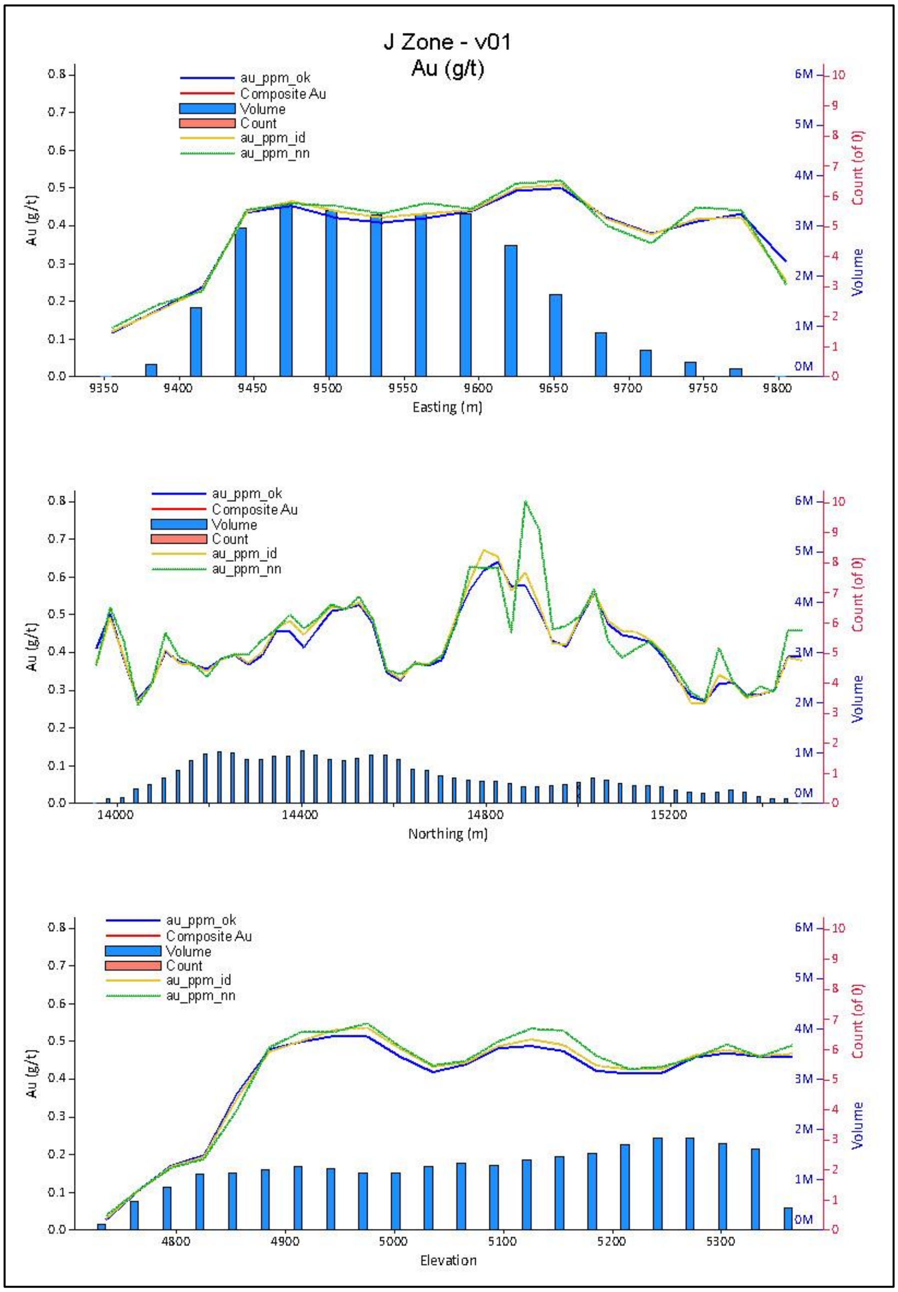
Table 14-52 presents the comparison of the mean gold, copper, and silver ordinary kriging (OK) estimated grades comparing to inverse distance powered two (ID^2) interpolated mean grades and nearest neighbour (NN) assigned grade by mineralized domain. In general, the achieved mean grade is reasonable.

Table 14-52: Mean Grade Comparison – J Zone

Rocktype	Au (g/t) Composite	Au (g/t) Ok	Au (g/t) ID	Au (g/t) NN	Cu (%) Composite	Cu (%) Ok	Cu (%) ID	Cu (%) NN	Ag (g/t) Composite	Ag (g/t) Ok	Ag (g/t) ID	Ag (g/t) NN
LG Halo	0.113	0.122	0.123	0.121	0.024	0.026	0.026	0.026	0.36	0.370	0.370	0.371
v01	0.473	0.427	0.435	0.442	0.046	0.042	0.042	0.042	0.86	0.833	0.841	0.853
v01 HG	1.212	1.129	1.106	1.101	0.059	0.059	0.059	0.058	1.13	1.150	1.144	1.137
v02	0.305	0.258	0.252	0.259	0.073	0.058	0.058	0.058	0.93	0.873	0.867	0.864
v03	0.362	0.343	0.337	0.330	0.049	0.051	0.051	0.051	0.70	0.724	0.722	0.717
v04	0.343	0.361	0.361	0.376	0.056	0.055	0.054	0.055	0.77	0.740	0.744	0.758
v04 HG	1.598	1.475	1.430	1.368	0.057	0.061	0.061	0.064	1.06	0.999	0.993	1.055
v05	0.389	0.261	0.265	0.287	0.062	0.065	0.066	0.072	1.12	1.107	1.135	1.233
v05 HG	0.590	0.571	0.568	0.574	0.193	0.188	0.188	0.196	2.86	2.764	2.756	2.880
v06	0.395	0.381	0.382	0.383	0.066	0.067	0.067	0.068	0.82	0.817	0.812	0.819
v07	0.290	0.286	0.282	0.292	0.066	0.067	0.066	0.068	0.60	0.591	0.583	0.624
v08	0.552	0.508	0.505	0.510	0.054	0.054	0.053	0.052	1.05	1.089	1.085	1.067
v10	0.414	0.373	0.374	0.377	0.070	0.071	0.071	0.073	1.10	1.093	1.092	1.125
v11	0.333	0.347	0.349	0.359	0.055	0.061	0.060	0.061	0.83	1.012	1.014	0.993
v12	0.203	0.172	0.171	0.174	0.092	0.090	0.088	0.087	0.69	0.698	0.704	0.694
v13	0.296	0.215	0.228	0.238	0.077	0.076	0.076	0.079	0.64	0.600	0.597	0.597
v15	0.485	0.418	0.420	0.424	0.072	0.070	0.069	0.070	1.38	1.383	1.370	1.324
v16	0.284	0.299	0.297	0.306	0.045	0.042	0.042	0.042	0.48	0.466	0.460	0.457
v17	0.191	0.190	0.190	0.187	0.043	0.043	0.042	0.043	0.51	0.502	0.503	0.510
v18	0.183	0.211	0.213	0.214	0.039	0.038	0.036	0.038	0.38	0.486	0.464	0.500

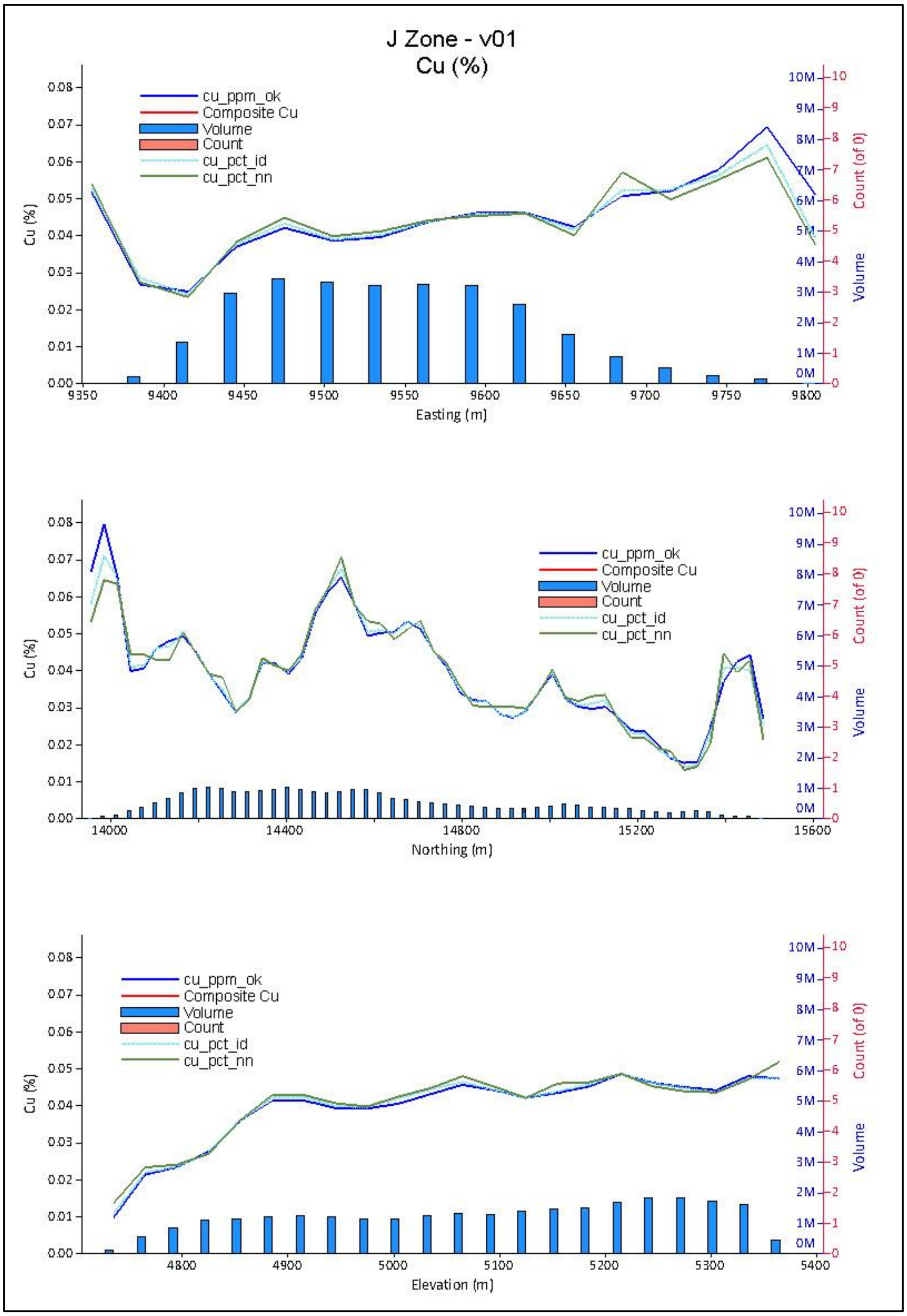
Swath plots were reviewed by northing easting and elevation. The distribution of gold, copper and silver composite values and interpolated block grades were compared to the OK, ID^2, and NN grades. Variables estimated with OK and ID^2 agree well in general, and no major spatial bias was observed. Figures 14-16 and Figure 14-17 present the swath plots for gold and copper, respectively, for domain v01. Figure 14-18 and Figure 14-19 present the swath plots for gold and copper, respectively, for domain v08.

Figure 14-16: Swath Plot for Gold, by Easting, Northing and Elevation; Domain v01 – J Zone



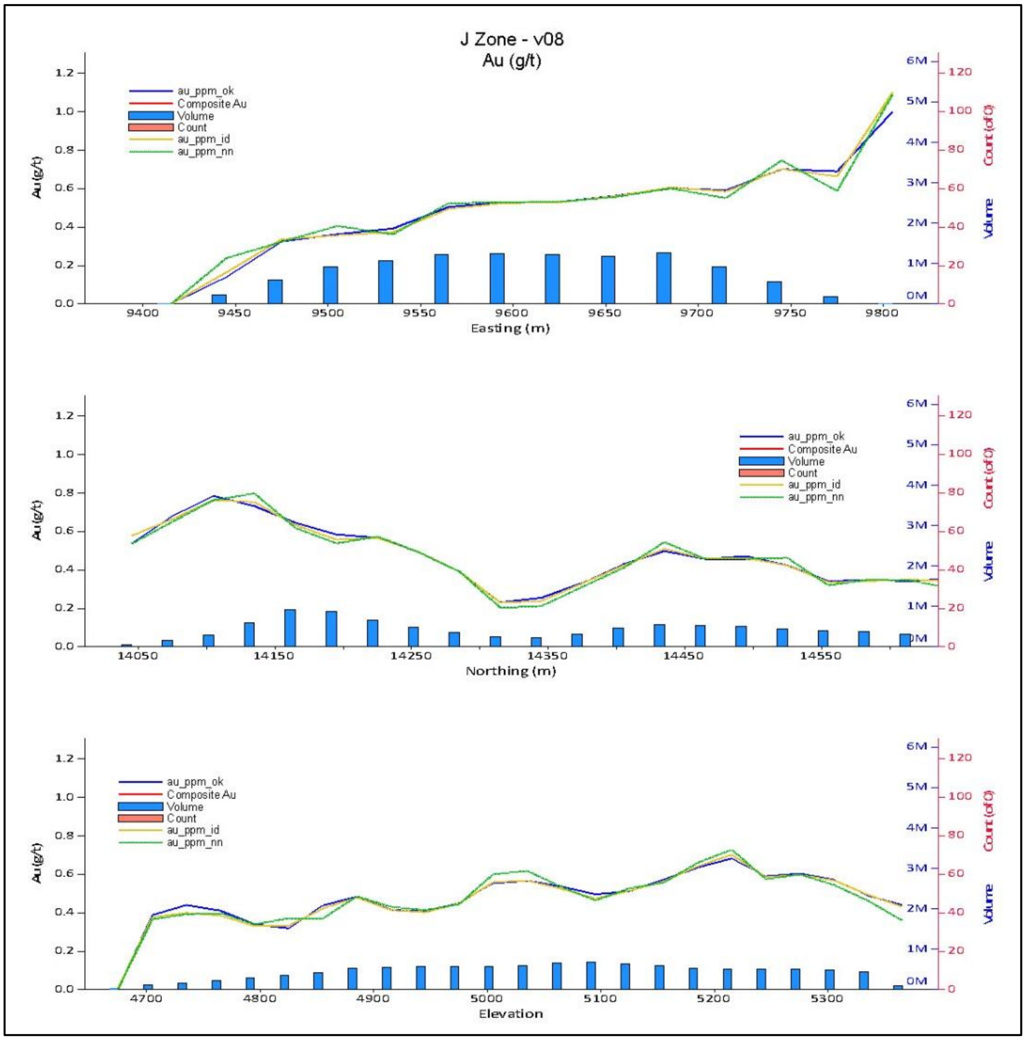
Source: AGP (2023)

Figure 14-17-: Swath Plot for Copper, by Easting, Northing and Elevation; Domain v08 – J Zone



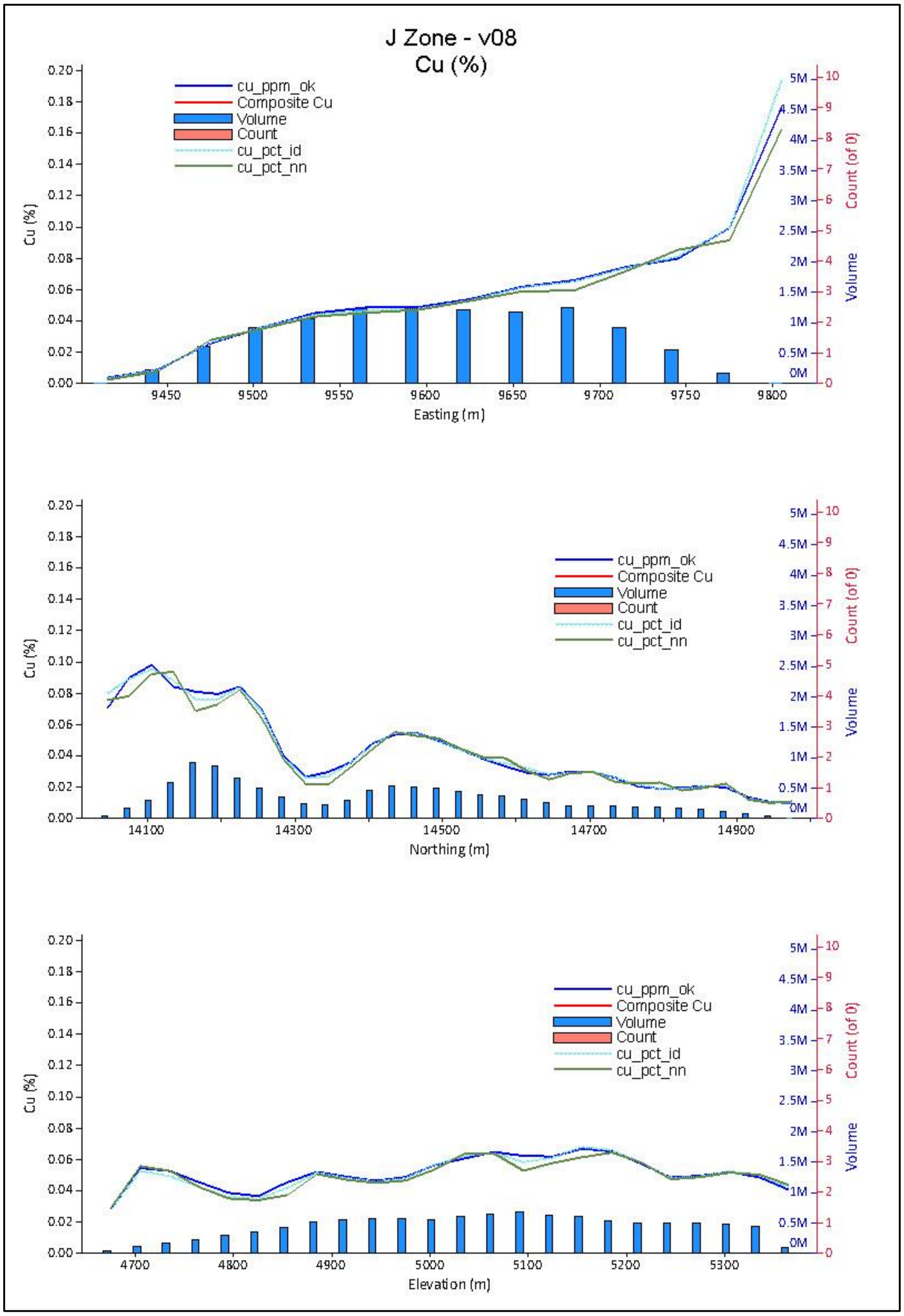
Source: AGP (2023)

Figure 14-18: Swath Plot for Gold, by Easting, Northing and Elevation; Domain v08 – J Zone



Source: AGP (2023)

Figure 14-19: Swath Plot for Copper, by Easting, Northing and Elevation; Domain v08 – J Zone



Source: AGP (2023)

14.7 X22 Zone

14.7.1 Interpretation

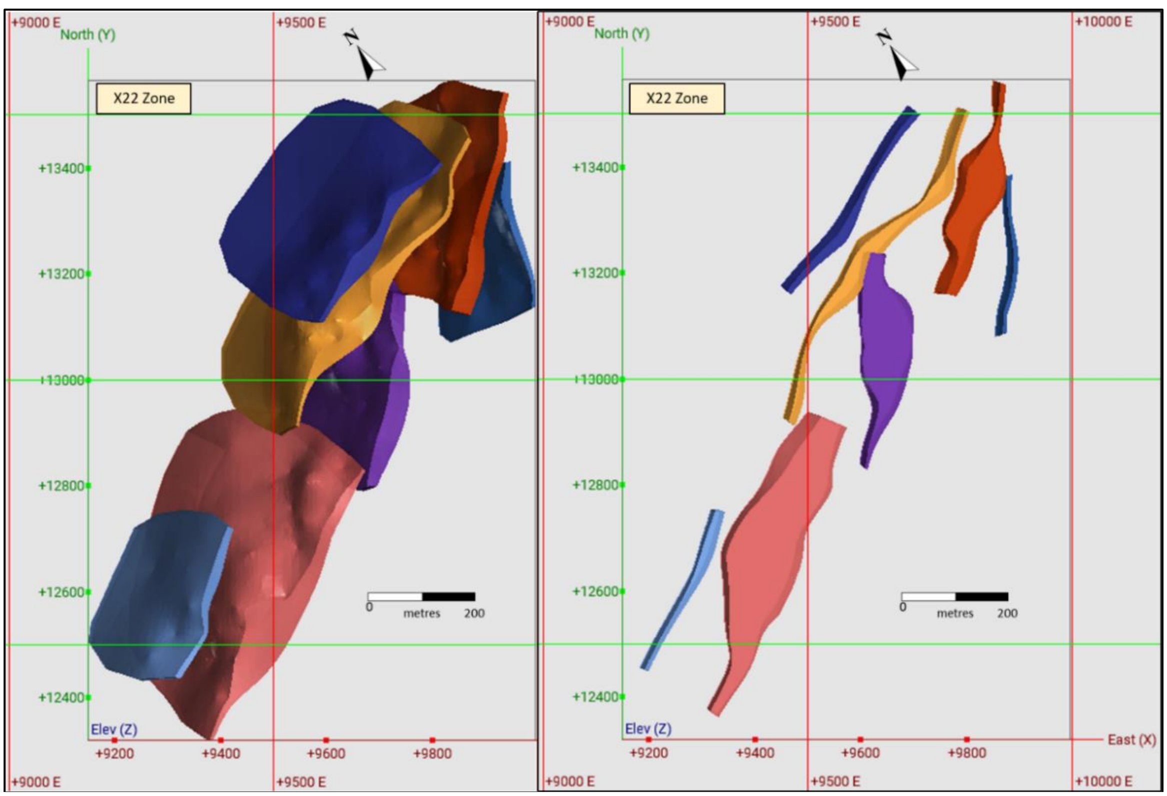
The mineralized domains at X22 Zone were interpreted by Troilus personnel. The interpreted wireframes were completed using Leapfrog Geo where grades were captured using a minimum grade of 0.3 g/t AuEQ and with a minimum thickness of 5 m. The AuEQ formula used for the gold equivalent was updated in 2023 as follows:

$$AuEQ = Au\ grade + (1.5628 * Cu\ grade) + (0.0128 * Ag\ grade)$$

All mineralized domain envelopes were created above pre-mining topography and clipped to the overburden bottom surface and then, to the pit topography. A total of seven 3D wireframes described the mineralized domains in X22 Zone. A surrounding, low-grade, domain was created based on grades greater than 0.07 g/t Au. The mineralization is disseminated and shows enrichment in what could be described as mineralized corridors without sharp boundaries. AGP considers the wireframes suitable to estimate resources.

Figure 14-20 shows the mineralized domain wireframes for the X22 Zone in 3D plan view and plan view section (5,190 m; 20 m viewing corridor). Table 14-53 shows the mineralized domains and the Domain Code

Figure 14-20: Mineralized Zones - 3D plan view and plan view section (5200 m; 20 m viewing corridor) X22 Zone



Source: AGP (2023)

Table 14-53: Mineralized Zones and Domain Codes – X22 Zone

Mineralized Zone	Domain Code
1401	1401
1402	1402
1403	1403
1404	1404
1405	1405
1406	1406
1407	1407
Low Grade Halo	99

14.7.2 Exploratory Data Analysis

Raw Assays

The drill hole database for the mineralized domains in the X22 Zone, consists of 16,525 raw assay values for each metal: gold, copper, and silver. The assay values reported below detection limit were assigned half the detection limit for statistical analysis and grade estimation. Any missing values were assigned half the detection limit.

Table 14-54 to Table 14-56 presents the descriptive statistics of the drill holes in the in the X22 Zone within the mineralized domains.

Table 14-54: Descriptive Statistics for Gold assays (g/t) by the Mineralized Domains – X22 Zone

Mineralized Zone	Count	Min	Max	Mean	Median	StDev	CV
1401	2285	0	126.50	0.52	0.17	3.07	5.91
1402	1731	0	11.60	0.41	0.27	0.57	1.39
1403	1301	0	23.90	0.67	0.23	1.62	2.43
1404	1135	0	133.00	0.64	0.16	5.21	8.20
1405	168	0.006	7.47	0.35	0.17	0.74	2.14
1406	274	0	15.55	0.36	0.13	1.19	3.32
1407	162	0	6.53	0.34	0.16	0.64	1.88
LG (99)	10253	0	32.40	0.10	0.03	0.59	5.88

Table 14-55: Descriptive Statistics for Copper assays (%) by the Mineralized Domains – X22 Zone

Mineralized Zone	Count	Min	Max	Mean	Median	StDev	CV
1401	2285	0	6.19	0.08	0.03	0.25	3.29
1402	1731	0	0.51	0.06	0.05	0.05	0.91
1403	1301	0	0.69	0.01	0.00	0.04	2.96
1404	1135	0	1.15	0.04	0.02	0.08	1.75
1405	168	0	0.34	0.01	0.00	0.04	2.71
1406	274	0	0.47	0.03	0.01	0.05	1.59
1407	162	0	0.72	0.05	0.03	0.09	1.75
LG (99)	10253	0	2.37	0.01	0.00	0.03	3.09

Table 14-56: Descriptive Statistics for Silver assays (g/t) by the Mineralized Domains – X22 Zone

Mineralized Zone	Count	Min	Max		Mean	Median	StDev	CV
1401	2284	0.1	883.00		2.43	0.50	21.51	8.86
1402	1728	0.1	26.90		0.75	0.50	1.32	1.75
1403	1116	0.1	46.30		0.56	0.25	2.23	3.98
1404	1119	0.1	70.20		1.11	0.28	3.77	3.40
1405	162	0.1	10.10		0.46	0.25	0.95	2.09
1406	252	0.1	12.60		0.66	0.25	1.16	1.76
1407	161	0.1	11.60		0.86	0.25	1.48	1.71
LG (99)	9703	0.1	126.10		0.39	0.25	1.73	4.45

Capping Analysis

Capping analysis was carried out on metal grades within each mineralized domain for gold, copper and silver by disintegration analysis, histogram, and probability plots. Capping was applied to metal grades where necessary.

Table 14-57 presents the selected capping levels gold and silver capping levels by domain. Descriptive statistics for capped gold and silver assay values are presented in Table 14-58 to Table 14-60, respectively.

Table 14-57: Capping Levels by Domain – X22 Zone

Domain	Au (g/t)	Loss (%)	Cu (%)	Loss (%)	Ag (g/t)	Loss (%)
1401	6.7 (12)	20.0	1.10 (16)	13.0	22.2 (23)	37.0
1402	7.2 (1)	0.6	-	-	11.5 (3)	3.0
1403	10.4 (7)	4.7	0.11 (20)	14.0	4.9 (10)	21.0
1404	4.1 (14)	45.0	0.37 (7)	5.7	8.7 (16)	21.0
1405	3.2 (1)	7.3	0.06 (9)	34.0	3.2 (2)	12.0
1406	2.7 (4)	24.0	0.23 (1)	3.1	4.4 (2)	6.4
1407	3.0 (1)	6.4	0.35 (3)	7.8	5.3 (2)	8.6
99 (LG)	3.2 (18)	14.0	0.21 (22)	5.0	11.3 (9)	6.8

(x) – number of values capped

Table 14-58: Descriptive Statistics for Capped Au (g/t) Assays by Mineralized Domain – X22 Zone

Mineralized Zone	Count	Min	Max	Mean	Median	StDev	CV
1401	2285	0	126.50	0.52	0.17	5.91	2285
1402	1731	0	11.60	0.41	0.27	1.39	1731
1403	1301	0	23.90	0.67	0.23	2.43	1301
1404	1135	0	133.00	0.64	0.16	8.20	1135
1405	168	0.006	7.47	0.35	0.17	2.14	168
1406	274	0	15.55	0.36	0.13	3.32	274
1407	162	0	6.53	0.34	0.16	1.88	162
LG (99)	10253	0	32.40	0.10	0.03	5.88	10253

Table 14-59: Descriptive Statistics for Capped Cu (%Cu) Assays by Mineralized Domain – X22 Zone

Mineralized Zone	Count	Min	Max	Mean	Median	StDev	CV
1401	2285	0	1.10	0.07	0.03	0.13	1.99
1402	1731	0	0.51	0.06	0.05	0.05	0.91
1403	1301	0	0.11	0.01	0.00	0.02	1.87
1404	1135	0	0.37	0.04	0.02	0.05	1.27
1405	168	0	0.06	0.01	0.00	0.02	1.64
1406	274	0	0.23	0.03	0.01	0.04	1.38
1407	162	0	0.35	0.05	0.03	0.06	1.38
LG (99)	10253	0	0.21	0.01	0.00	0.02	1.86

Table 14-60: Descriptive Statistics for Capped Ag (g/t) Assays by Mineralized Domain – X22 Zone

Mineralized Zone	Count	Min	Max	Mean	Median	StDev	CV
1401	2284	0.1	22.20	1.53	0.50	3.22	2.11
1402	1728	0.1	11.50	0.74	0.50	1.02	1.39
1403	1116	0.1	7.90	0.45	0.25	0.73	1.61
1404	1119	0.1	8.70	0.94	0.28	1.64	1.75
1405	162	0.1	3.20	0.40	0.25	0.53	1.31
1406	252	0.1	4.40	0.62	0.25	0.85	1.39
1407	161	0.1	5.30	0.79	0.25	1.03	1.31
LG (99)	9703	0.1	11.60	0.36	0.25	0.58	1.61

Composites

The capped assays were composited to two metre lengths for those assays captured within the mineralized domains, starting at domain boundary. Composites were adjusted across the intersection of the domain.

Table 14-61 to Table 14-63 present the descriptive statistics for the 2 m capped composite values for gold, copper, and silver by domain, respectively, in the X22 Zone.

Table 14-61: Descriptive Statistics for Capped Au (g/t) Composite Values by Mineralized Domain – X22 Zone

Mineralized Zone	Count	Min	Max	Mean	Median	StDev	CV
1401	1166	0	6.70	0.41	0.21	0.64	1.56
1402	866	0	3.91	0.41	0.30	0.41	1.01
1403	832	0	10.40	0.48	0.17	0.90	1.89
1404	608	0	3.48	0.33	0.18	0.45	1.39
1405	124	0	1.69	0.22	0.10	0.36	1.59
1406	143	0	1.66	0.25	0.15	0.32	1.29
1407	85	0	2.60	0.31	0.17	0.39	1.26
LG (99)	5579	0	2.07	0.06	0.02	0.15	2.53

Table 14-62: Descriptive Statistics for Capped Cu (%) Composite Values by Mineralized Domain – X22 Zone

Mineralized Zone	Count	Min	Max	Mean	Median	StDev	CV
1401	1166	0	1.10	0.07	0.03	0.11	1.70
1402	866	0	0.44	0.06	0.05	0.05	0.80
1403	832	0	0.11	0.01	0.00	0.02	1.90
1404	608	0	0.22	0.04	0.03	0.04	1.00
1405	124	0	0.06	0.01	0.00	0.01	1.75
1406	143	0	0.14	0.03	0.02	0.03	1.14
1407	85	0	0.28	0.04	0.03	0.05	1.23
LG (99)	5579	0	0.19	0.01	0.00	0.02	1.68

Table 14-63: Descriptive Statistics for Capped Ag (g/t) Composite Values by Mineralized Domain – X22 Zone

Mineralized Zone	Count	Min	Max	Mean	Median	StDev	CV
1401	1157	0.04	22.20	1.52	0.57	2.72	1.79
1402	862	0.10	11.50	0.74	0.53	0.85	1.15
1403	688	0	5.45	0.35	0.25	0.56	1.58
1404	558	0	8.70	0.93	0.49	1.31	1.40
1405	120	0	1.85	0.27	0.25	0.37	1.39
1406	120	0.04	4.40	0.59	0.25	0.73	1.23
1407	85	0	4.20	0.75	0.38	0.87	1.17
LG (99)	4193	0	8.51	0.45	0.25	0.63	1.39

Variography

Spatial analysis was performed on 2 m capped composite gold, silver, and copper data within the mineralized domains and the variogram parameters were assigned to each mineralized domain. Table 14-64 to Table 14-66 presents the variogram parameters for the X22 Zone for gold, silver, and copper values by domain.

Table 14-64: Variogram Parameters for Gold by Domain –X22 Zone

Domain	Nugget	Structures Sill=1.0	Dip (°)	Dip Az. (°)	Pitch (°)	Major (m)	Semi-Major (m)	Minor (m)	Anisotropy
1401	0.3	0.10	58	293	70	70	75	2	Spheroidal
		0.60	58	293	70	100	125	10	Spheroidal
1402	0.3	0.10	58	293	70	70	75	2	Spheroidal
		0.60	58	293	70	100	125	10	Spheroidal
1403	0.3	0.36	58	293	155	70	75	2	Spheroidal
		0.34	58	293	155	100	125	4	Spheroidal
1404	0.3	0.07	58	293	156	70	75	2	Spheroidal
		0.63	58	293	156	91	130	5	Spheroidal
1405	0.3	0.04	58	293	0	33	36	2	Spheroidal
		0.66	58	293	0	45	62	10	Spheroidal
1406	0.3	0.10	58	293	70	70	75	2	Spheroidal
		0.60	58	293	70	100	125	10	Spheroidal
1407	0.3	0.10	58	293	70	70	75	2	Spheroidal
		0.60	58	293	70	100	125	10	Spheroidal
99 (LG)	0.2	0.07	58	293	70	70	75	2	Spheroidal
		0.73	58	293	70	70	120	5	Spheroidal

Az = Azimuth

Table 14-65: Variogram Parameters for Copper by Domain –X22 Zone

Domain	Nugget	Structures Sill=1.0	Dip (°)	Dip Az. (°)	Pitch (°)	Major (m)	Semi-Major (m)	Minor (m)	Anisotropy
1401	0.2	0.35	58	293	70	57	92	3	Spheroidal
		0.45	58	293	70	80	115	14	Spheroidal
1402	0.15	0.52	58	293	70	65	82	3	Spheroidal
		0.33	58	293	70	115	100	14	Spheroidal
1403	0.2	0.34	58	293	155	74	86	3	Spheroidal
		0.46	58	293	155	89	92	9	Spheroidal
1404	0.2	0.34	58	293	156	74	86	3	Spheroidal
		0.46	58	293	156	89	92	9	Spheroidal
1405	0.2	0.46	58	293	0	43	72	2.5	Spheroidal
		0.34	58	293	0	78	140	9	Spheroidal
1406	0.15	0.05	58	293	70	43	73	2.5	Spheroidal
		0.80	58	293	70	78	120	9	Spheroidal
1407	0.15	0.05	58	293	70	43	73	2.5	Spheroidal
		0.80	58	293	70	78	120	9	Spheroidal
99 (LG)	0.15	0.27	58	293	70	57	92	3	Spheroidal
		0.58	58	293	70	90	115	14	Spheroidal

Az = Azimuth

Table 14-66: Variogram Parameters for Silver by Domain –X22 Zone

Domain	Nugget	Structures Sill=1.0	Dip (°)	Dip Az. (°)	Pitch (°)	Major (m)	Semi-Major (m)	Minor (m)	Anisotropy
1401	0.3	0.08	58	293	70	70	60	2	Spheroidal
		0.60	58	293	70	90	95	10	Spheroidal
1402	0.3	0.10	58	293	70	70	75	2	Spheroidal
		0.60	58	293	70	100	63.6	10	Spheroidal
1403	0.3	0.28	58	293	155	50	26	2	Spheroidal
		0.34	58	312	168	60	74	4	Spheroidal
1404	0.2	0.01	58	312	168	60	83	2	Spheroidal
		0.63	58	293	156	105.5	68.64	5	Spheroidal
1405	0.3	0.70	58	293	0	33	120	2	Spheroidal
1406	0.3	0.10	58	293	70	70	75	2	Spheroidal
		0.60	58	293	70	100	125	10	Spheroidal
1407	0.3	0.10	58	293	70	70	75	2	Spheroidal
		0.60	58	293	70	100	125	10	Spheroidal
99 (LG)	0.2	0.07	58	293	70	40	62	2	Spheroidal
		0.73	58	293	70	54	120	5	Spheroidal

Az = Azimuth

14.7.3 Block Model

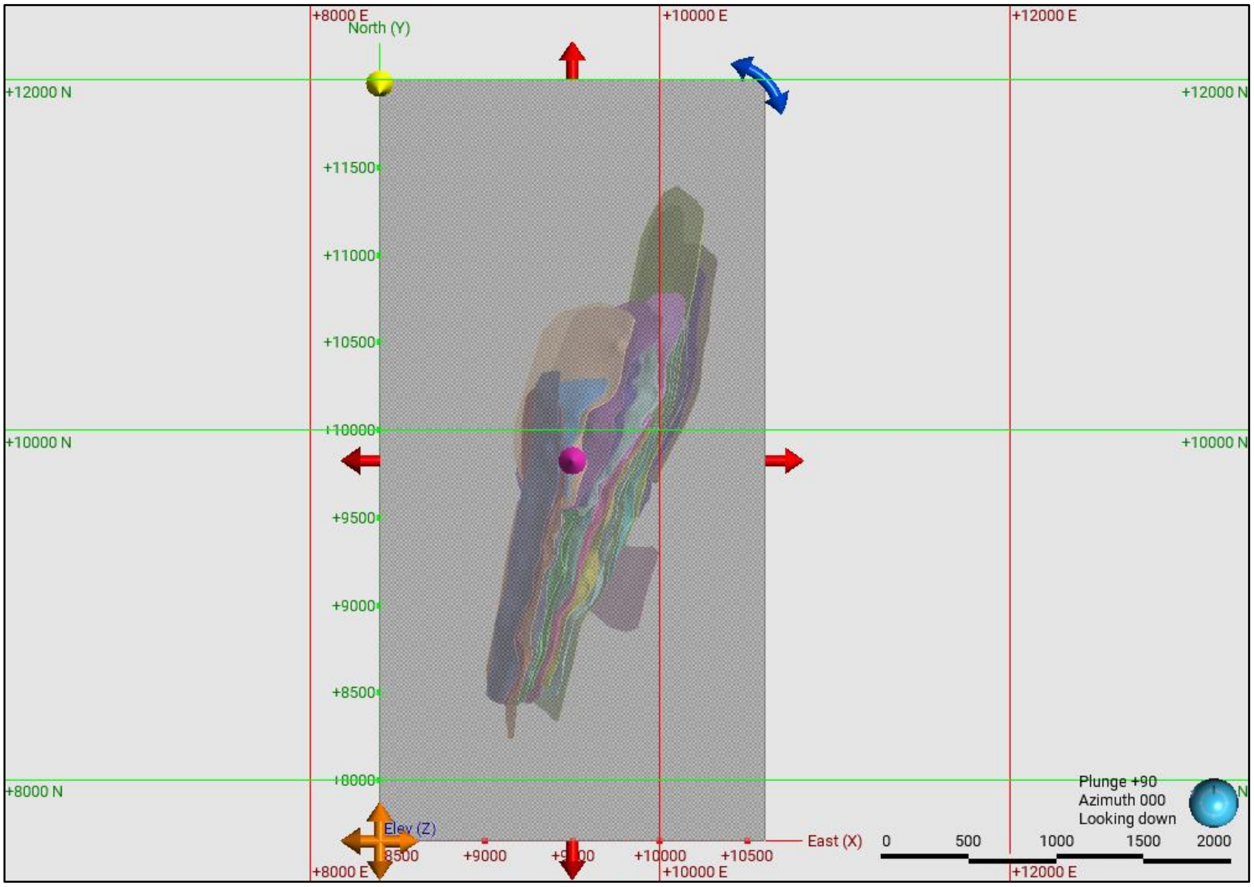
Block Model Parameters

The block model for the X22 Zone deposit was set up with a block matrix of 5 m long by 5 m wide by 5 m high and was created using Leapfrog Edge resource software. The block matrix was defined based on current drill hole spacing and on engineering considerations for an open pit operation and is considered suitable this purpose. The block model is in mine grid coordinates and is not rotated. Table 14-67 summarizes the block model parameters and Figure 14-21 presents the block model over the interpreted mineralized domains for the X22 Zone.

Table 14-67: Block Model Parameters – X22 Zone

Domain	Minimum	Maximum
Easting	8900 mE	10160 mE
Northing	12050 mN	13865 mN
Maximum Elevation	4500 m	5500 m
Rotation Angle	No rotation°	
Block Size (X, Y, Z in metres)	5 x 5 x 5	
Number of blocks in the X direction	252	
Number of blocks in the Y direction	363	
Number of blocks in the Z direction	200	

Figure 14-21: Block Model Extents – X22 Zone



Source: AGP (2023)

The block model extents cover the entire X22 Zone and is extended on all four sides beyond the interpreted mineralized domains. The block model is a whole block model where blocks are assigned a specific rock type code. Any block centroid within the mineralized domain wireframe was assigned that code.

Block model attributes in the block model includes:

- domain code
- metal grades for gold, copper, silver, and calculated gold-equivalent grades for estimated blocks
- classification
- distance to the nearest composite
- average distance of estimated composites
- number of composites used in estimation of a block
- number of drill holes used in estimation of a block
- pass number

- lithology code
- density

Estimation Parameters and Interpolation Strategy

The metal grades were interpolated in three passes using the 2 m capped composites by domain. The metal grades were interpolated using OK interpolation method. Variogram parameters for each metal was used in each of these passes and used variable orientation aligned to the mineralized domain wireframe. ID^2 and NN interpolations were also run for validation purposes.

Each pass required the same minimum and maximum number of composites for each domain. A maximum of three composites per drill hole was used. Table 14-68 shows estimation parameters for each pass used to estimate metal grades.

Table 14-68: Estimation Parameters – X22 Zone

Pass	Min No Composites	Max No Composites	Max No. Composites per Hole	Min. No. of Drill Holes
Pass 1	4	18	3	2
Pass 2	4	18	3	2
Pass 3	3	18	3	1

Due to the sinuous nature of the interpreted mineralized domains, search ellipses were allowed a variable orientation that follow the trend of each domain. Each pass increased the search ellipse where Pass 2 was doubled that of Pass 1 and Pass 3 was approximately double that of Pass 2. Hard boundaries were kept between all domains and blocks within each domain were estimated only be composites within the domain wireframe.

Table 14-69 shows search ellipse parameters for each subdomain used to estimate metal grades.

Table 14-69: Search Ellipse Parameters for Gold, Copper, and Silver– X22 Zone

All Domains	Orientation	Major (m)	Semimajor (m)	Minor (m)	Search
Pass 1	Variable Orientation	75	50	10	Ellipsoidal
Pass 2	Variable Orientation	130	130	20	Ellipsoidal
Pass 3	Variable Orientation	200	150	30	Ellipsoidal

Block Model Validation

The block model was validated using the following methods:

- statistical comparison of resource assay and block grade distributions
- visual inspection and comparison of block grades with composite and assay grades
- inspection of swath plots with composites and block grades elevations and northings

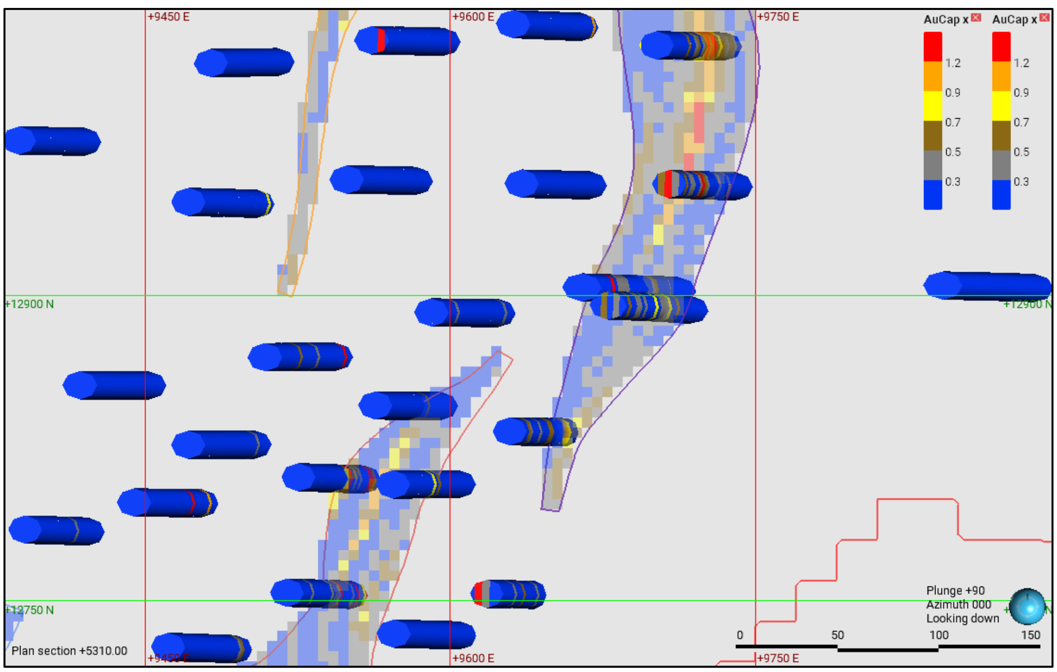
Table 14-70 shows the comparison of mean gold grades by domain for estimated blocks to the ID^2 and NN interpolations and the 2 m composite values. Variables estimated with OK and ID^2 generally agree with some smoothing of grades noted but no major spatial bias was observed.

Table 14-70: Comparison of Mean Gold Grades (g/t Au) – SW Zone

Domain	OK	ID^2	NN	2 m Comps
1401	0.45	0.44	0.46	0.41
1402	0.43	0.42	0.41	0.41
1403	0.59	0.58	0.63	0.52
1404	0.36	0.36	0.39	0.33
1405	0.26	0.26	0.27	0.27
1406	0.26	0.26	0.41	0.25
1407	0.33	0.34	0.39	0.31
LG (99)	0.08	0.08	0.08	0.08

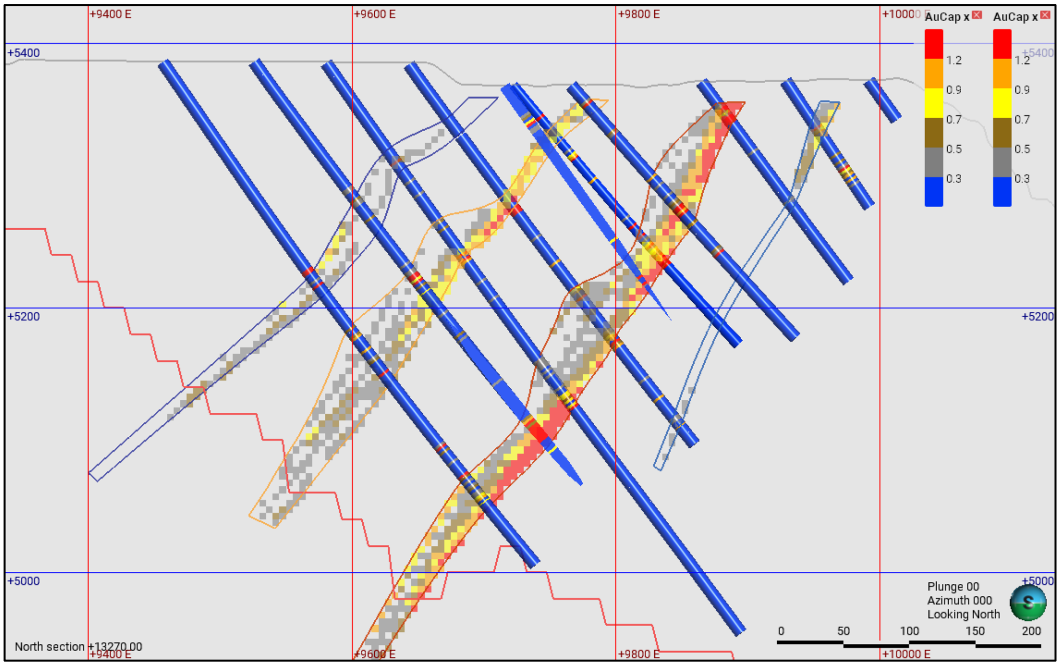
The block model grades, and the composites grades were visually inspected on plan and cross sections. Composite grades honour the block grades well within the mineralized domains. Figure 14-22 and Figure 14-23 present selected plan section and cross section views for the X22 Zone (plan view elevation 5310 m and cross section 13270 mN), respectively.

Figure 14-22: Plan-section at Elevation 5310 m – X22 Zone



Source: AGP (2023)

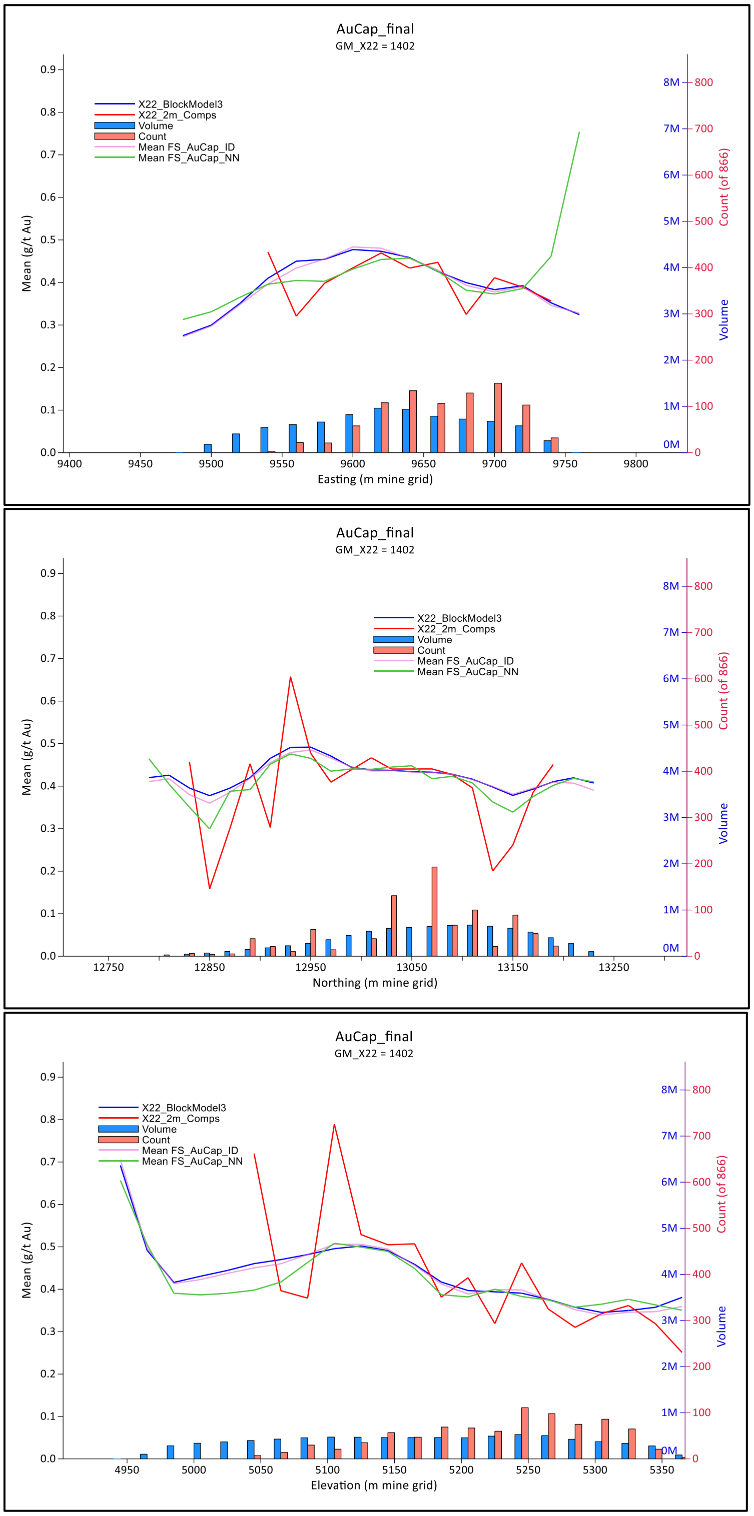
Figure 14-23: Cross-section 13270 mN – X22 Zone



Source: AGP (2023)

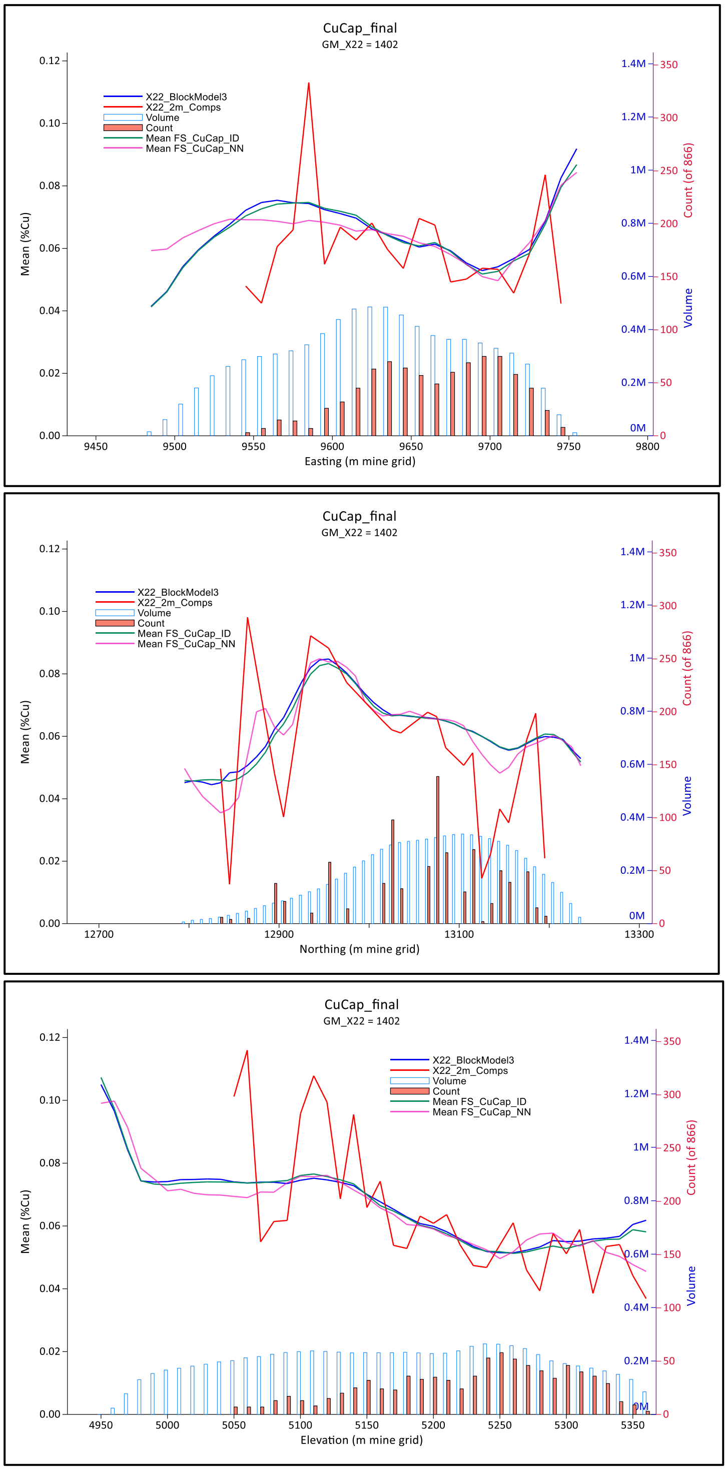
Swath plots were reviewed by northing, easting, and elevation. The distribution of gold, copper and silver composite values and interpolated block grades were compared to the OK, ID^2 and NN grades. No issues were found with the distribution of interpolated grades. Figure 14-24 and Figure 14-25 present the swath plots for gold and copper, respectively, for domain 1402 by easting, northing and elevation.

Figure 14-24: Swath Plot for Gold, by Northing and Elevation; Domain 1402 – X22 Zone



Source: AGP (2023)

Figure 14-25: Swath Plot for Copper, by Northing and Elevation; Domain 1402 – X22 Zone



Source: AGP (2023)

14.8 SW Zone

14.8.1 Interpretation

The mineralized domains at SW Zone were interpreted by Troilus personnel. The interpreted wireframes were completed using Leapfrog Geo where grades were captured using a minimum grade of 0.3 g/t AuEQ with a minimum thickness of 3.0 m applied for all domains.

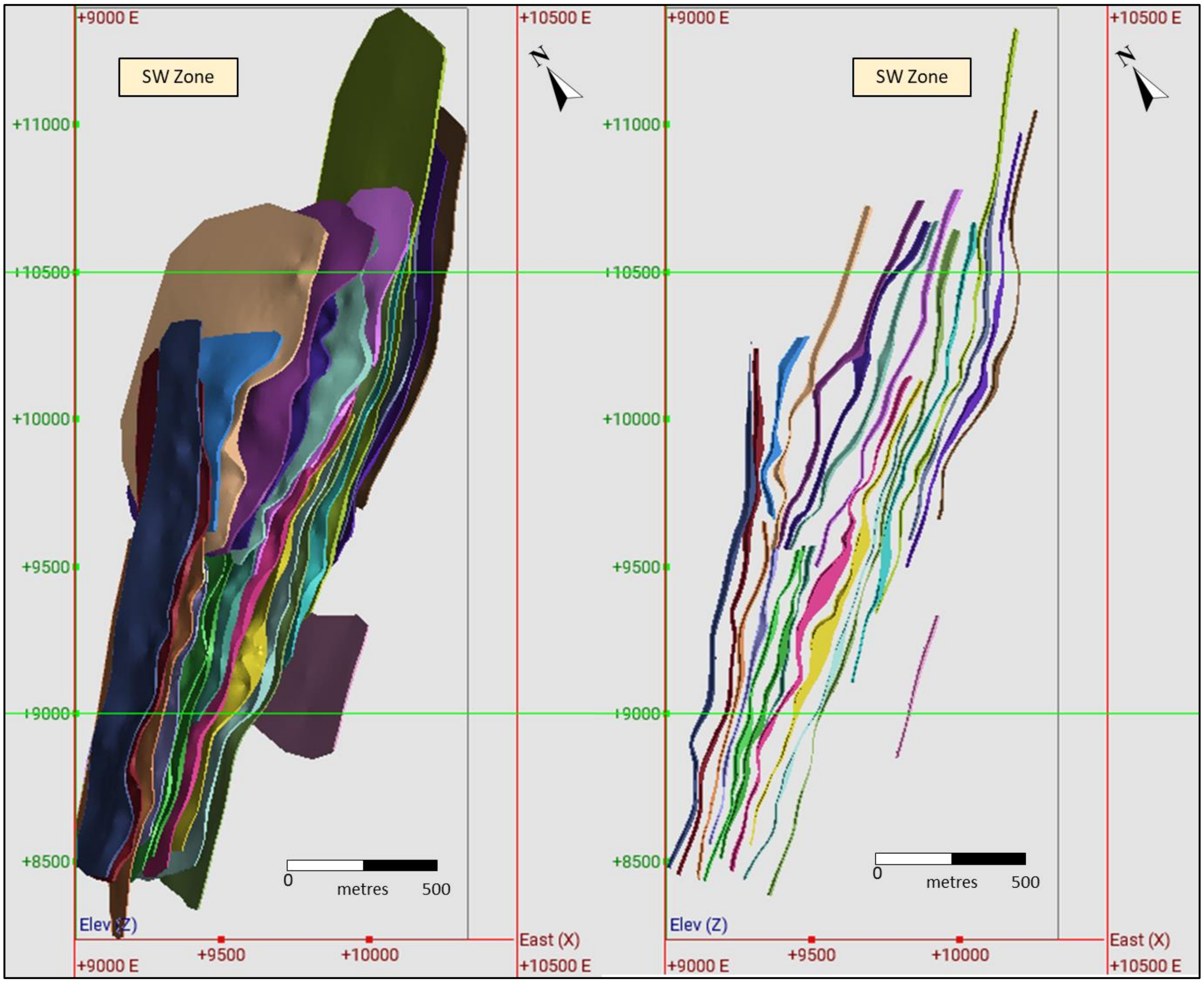
The AuEQ formula used for the gold equivalent formula for the SW Zone from the PEA Report (AGP, 2020) as follows:

$$AuEQ = Au\ grade + (1.2768 * Cu\ grade) + (0.0106 * Ag\ grade)$$

All mineralized domain envelopes were created above pre-mining topography and clipped to the overburden bottom surface and then, to the pit topography. A total of 23 3D wireframes described the mineralized domains in SW Zone. A surrounding, low-grade domain was created based on grades greater than 0.07 g/t Au. The mineralization is disseminated and shows enrichment in what could be described as mineralized corridors without sharp boundaries. The wireframes provided were mostly intended to limit grade smearing between high grade zones into lower grade material (and vice-versa). AGP considers the wireframes suitable to estimate resources.

Figure 14-26 shows the mineralized domain wireframes for the SW Zone in 3D plan view and plan view section (5,190 m; 20 m viewing corridor).

Figure 14-26: Mineralized Zones; 3D plan view and plan view section (5190 m; 20 m viewing corridor) SW Zone



Source: AGP (2023)

14.8.2 Exploratory Data Analysis

Raw Assays

The drill hole database for the mineralized domains in the SW Zone, consists of 75,632 assay values within all mineralized domains for: gold, copper, and silver. The assay values reported below detection limit were assigned half the detection limit for statistical analysis and grade estimation. Any missing values were assigned half the detection limit.

Table 14-71 to Table 14-73 presents the descriptive statistics of the drill holes for gold, copper, and silver, respectively, in the in the SW Zone within the mineralized domains.

Table 14-71: Descriptive Statistics for Gold assays (g/t) by the Mineralized Domains – SW Zone

Mineralized Zone	Count	Min	Max	Mean	Median	StDev	CV
199	535	0.001	18.15	0.44	0.19	1.29	2.95
200	729	0.003	13.15	0.45	0.21	0.91	2.01
201	1364	0.003	15.10	0.34	0.16	0.86	2.56
202	2967	0.003	27.40	0.48	0.22	1.16	2.39
203	2062	0.003	46.30	0.51	0.20	1.48	2.90
204	330	0.003	4.36	0.22	0.09	0.47	2.16
205	768	0.003	8.85	0.33	0.20	0.61	1.84
206	322	0.003	10.95	0.43	0.09	1.02	2.38
207	288	0.003	6.65	0.40	0.15	0.74	1.84
208	995	0.003	2320.00	2.75	0.11	73.58	26.76
209	370	0.001	12.70	0.23	0.07	0.81	3.53
210	583	0.003	11.15	0.27	0.15	0.57	2.13
211	1263	0.003	18.10	0.39	0.15	1.03	2.62
212	774	0.003	92.00	0.48	0.18	3.34	6.94
213	41	0.008	3.49	0.50	0.17	0.79	1.56
214	381	0.003	5.04	0.36	0.18	0.55	1.53
215	827	0.001	68.00	0.51	0.17	2.61	5.14
216	505	0.001	6.79	0.40	0.16	0.76	1.92
217	467	0.003	30.60	0.51	0.11	1.82	3.56
218	633	0.001	12.15	0.33	0.11	0.88	2.63
219	310	0.001	12.45	0.34	0.07	1.03	3.00
220	346	0.001	14.75	0.37	0.06	1.18	3.15
221	495	0.001	36.20	0.43	0.11	1.82	4.25
LG (99)	58277	0.001	1690	0.111	0.032	7.151	64.16

Table 14-72: Descriptive Statistics for Copper assays (%) by the Mineralized Domains – SW Zone

Mineralized Zone	Count	Min	Max	Mean	Median	StDev	CV
199	535	0	0.64	0.03	0.01	0.06	2.07
200	729	0	1.26	0.05	0.01	0.12	2.30
201	1364	0	1.66	0.06	0.03	0.09	1.45
202	2967	0	1.13	0.03	0.01	0.07	2.01
203	2062	0	2.11	0.07	0.03	0.12	1.69
204	330	0	0.47	0.05	0.03	0.06	1.29
205	768	0.001	0.68	0.07	0.04	0.08	1.23
206	322	0	0.54	0.02	0.00	0.04	2.89
207	288	0	0.29	0.01	0.00	0.02	3.14
208	995	0	0.42	0.04	0.01	0.06	1.49
209	370	0	0.54	0.05	0.02	0.07	1.52
210	583	0	1.05	0.06	0.03	0.09	1.48
211	1263	0	0.96	0.05	0.03	0.08	1.50
212	774	0	0.72	0.05	0.03	0.06	1.26
213	41	0.008	0.06	0.02	0.02	0.01	0.49
214	381	0	0.48	0.05	0.02	0.06	1.38
215	827	0	2.53	0.06	0.03	0.12	2.03
216	505	0	0.62	0.05	0.02	0.07	1.52
217	467	0	0.15	0.01	0.00	0.02	1.82
218	633	0.001	0.62	0.05	0.03	0.06	1.24
219	310	0	0.59	0.02	0.00	0.04	2.68
220	346	0	0.32	0.02	0.00	0.04	2.44
221	495	0	0.43	0.01	0.00	0.04	2.70
LG (99)	58277	0	2.59	0.02	0.01	0.03	1.73

Table 14-73: Descriptive Statistics for Silver assays (g/t) by the Mineralized Domains – SW Zone

Mineralized Zone	Count	Min	Max	Mean	Median	StDev	CV
199	535	0.10	86.10	1.84	0.50	5.79	3.15
200	729	0.10	42.60	1.18	0.50	2.50	2.12
201	1364	0.10	73.30	1.22	0.50	2.98	2.45
202	2967	0.10	44.50	0.57	0.25	1.29	2.26
203	2062	0.10	28.30	0.92	0.25	1.83	1.99
204	330	0.20	9.60	0.90	0.50	1.09	1.21
205	768	0.20	16.80	1.16	0.60	1.63	1.41
206	322	0.10	13.70	0.72	0.25	1.64	2.27
207	288	0.10	38.50	0.75	0.25	2.43	3.26
208	995	0.05	11.90	0.68	0.25	1.06	1.55
209	370	0.10	15.80	0.94	0.25	1.67	1.78
210	584	0	19.20	1.02	0.50	1.45	1.42
211	1263	0.05	30.40	0.86	0.25	1.62	1.89
212	774	0.10	55.00	1.48	0.60	3.75	2.54
213	41	0.25	218.00	22.15	2.00	40.91	1.85
214	381	0.10	7.40	0.59	0.25	0.71	1.19
215	828	0.00	1170.00	2.74	0.50	40.73	14.87
216	505	0.10	73.00	2.35	0.60	6.45	2.74
217	467	0.20	46.90	0.88	0.25	3.32	3.77
218	633	0.10	36.40	1.12	0.50	2.16	1.92
219	310	0.20	122.00	1.31	0.25	7.29	5.57
220	346	0.10	130.00	2.10	0.25	12.06	5.75
221	495	0.10	41.90	0.61	0.25	2.13	3.48
LG (99)	58312	0	149.00	0.46	0.25	1.25	2.74

Capping Analysis

Capping analysis was carried out on metal grades within each mineralized domain for gold, copper and silver by disintegration analysis, histogram, and probability plots. Capping was applied to metal grades where necessary.

Table 14-74 presents the selected capping levels gold and silver capping levels by domain. Descriptive statistics for capped gold and silver assay values are presented in Table 14-75 to Table 14-77, respectively.

Table 14-74: Capping Levels by Domain – SW Zone

Domain	Au (g/t)	Loss %)	Cu (%)	Loss (%)	Ag (g/t)	Loss (%)
199	4.3 (5)	16.0	0.29 (4)	5.0	22.0 (4)	15.0
200	8.0 (3)	2.8	0.60 (5)	5.0	24.8 (2)	3.3
201	6.1 (7)	5.4			19.0 (3)	5.6
202	14.6 (3)	1.3	0.49 (10)	2.9	16.6 (2)	2.2
203	12.6 (2)	5.6	1.04 (3)	1.1	15.4 (4)	2.1
204	3.4 (3)	2.0				
205	7.2 (2)	1.1			12.0 (2)	0.8
206	5.0 (5)	5.9	0.12 (8)	18.0	5.0 (6)	15.0
207	5.0 (2)	1.5	0.07 (3)	19.0	7.1 (2)	25.0
208	7.57 (6)	88.0			8.0 (3)	1.5
209	2.6 (3)	19.0			10.1 (2)	2.9
210	3.0 (1)	5.2			11.2 (1)	1.3
211	7.0 (5)	5.9	0.63 (3)	0.9	20.4 (1)	0.9
212	6.5 (2)	23.0			22.5 (3)	6.8
213	3.4 (1)	0.4			55.0 (2)	27.0
214						
215	8.0 (7)	20.0	0.80 (2)	3.5	17.1 (6)	54.0
216	5.5 (3)	1.0			18.1 (12)	20.0
217	5.8 (6)	18.0			7.8 (5)	23.0
218	3.2 (7)	14.0			18.8 (1)	2.5
219	3.2 (6)	19.0	0.16 (2)	10.0	9.5 (6)	36.0
220	3.7 (7)	18.0	0.19 (6)	6.4	6.9 (9)	66.0
221	5.5 (4)	19.0	0.21 (5)	7.5	10.6 (3)	14.0
99 (LG)	4.6 (24)	33	0.74(7)	0.4	27.3 (13)	1.4

(x) – number of values capped

Table 14-75: Descriptive Statistics for Capped Au (g/t) Assays by Mineralized Domain – SW Zone

Mineralized Zone	Count	Min	Max	Mean	Median	StDev	CV
199	535	0.001	4.30	0.37	0.19	0.61	1.66
200	729	0.003	8.00	0.44	0.21	0.75	1.72
201	1364	0.003	6.10	0.32	0.16	0.65	2.05
202	2967	0.003	14.60	0.48	0.22	1.05	2.20
203	2062	0.003	12.60	0.48	0.20	0.90	1.86
204	330	0.003	3.40	0.21	0.09	0.43	2.05
205	768	0.003	7.20	0.33	0.20	0.57	1.72
206	322	0.003	5.00	0.40	0.09	0.83	2.06
207	288	0.003	5.00	0.40	0.15	0.69	1.75
208	995	0.003	7.57	0.32	0.11	0.77	2.41
209	370	0.001	2.60	0.19	0.07	0.34	1.83
210	583	0.003	3.00	0.25	0.15	0.36	1.43
211	1263	0.003	7.00	0.37	0.15	0.74	2.00
212	774	0.003	6.50	0.37	0.18	0.61	1.64
213	41	0.008	3.40	0.50	0.17	0.78	1.55
214	381	0.003	5.04	0.36	0.18	0.55	1.53
215	827	0.001	8.00	0.41	0.17	0.89	2.19
216	505	0.001	5.50	0.39	0.16	0.73	1.86
217	467	0.003	5.80	0.42	0.11	0.90	2.15
218	633	0.001	3.20	0.29	0.11	0.50	1.75
219	310	0.001	3.20	0.28	0.07	0.56	2.02
220	346	0.001	3.70	0.31	0.06	0.68	2.23
221	495	0.001	5.00	0.34	0.11	0.65	1.90
LG (99)	58277	0.001	8.70	0.08	0.03	0.22	2.94

Table 14-76: Descriptive Statistics for Capped Cu (%Cu) Assays by Mineralized Domain – SW Zone

Mineralized Zone	Count	Min	Max	Mean	Median	StDev	CV
199	535	0	0.29	0.03	0.01	0.05	1.81
200	729	0	0.60	0.05	0.01	0.10	2.03
201	1364	0	1.66	0.06	0.03	0.09	1.45
202	2967	0	0.49	0.03	0.01	0.06	1.76
203	2062	0	1.04	0.07	0.03	0.11	1.57
204	330	0	0.47	0.05	0.03	0.06	1.29
205	768	0.001	0.68	0.07	0.04	0.08	1.23
206	322	0	0.12	0.01	0.00	0.02	1.99
207	288	0	0.07	0.01	0.00	0.01	1.74
208	995	0	0.42	0.04	0.01	0.06	1.49
209	370	0	0.54	0.05	0.02	0.07	1.52
210	583	0	1.05	0.06	0.03	0.09	1.48
211	1263	0	0.63	0.05	0.03	0.08	1.43
212	774	0	0.72	0.05	0.03	0.06	1.26
213	41	0.008	0.06	0.02	0.02	0.01	0.49
214	381	0	0.48	0.05	0.02	0.06	1.38
215	827	0	0.80	0.06	0.03	0.09	1.57
216	505	0	0.62	0.05	0.02	0.07	1.52
217	467	0	0.15	0.01	0.00	0.02	1.82
218	633	0.001	0.62	0.05	0.03	0.06	1.24
219	310	0	0.16	0.01	0.00	0.03	1.87
220	346	0	0.19	0.02	0.00	0.03	2.20
221	495	0	0.21	0.01	0.00	0.03	2.20
LG (99)	58277	0	0.74	0.02	0.01	0.03	1.55

Table 14-77: Descriptive Statistics for Capped Ag (g/t) Assays by Mineralized Domain – SW Zone

Mineralized Zone	Count	Min	Max	Mean	Median	StDev	CV
199	535	0.10	22.00	1.57	0.50	2.98	1.90
200	729	0.10	24.80	1.15	0.50	2.16	1.87
201	1364	0.10	19.00	1.15	0.50	1.89	1.65
202	2967	0.10	15.60	0.56	0.25	0.99	1.77
203	2062	0.10	15.40	0.90	0.25	1.61	1.78
204	330	0.20	9.60	0.90	0.50	1.09	1.21
205	768	0.20	12.00	1.15	0.60	1.56	1.36
206	322	0.10	5.00	0.61	0.25	1.02	1.66
207	288	0.10	7.10	0.64	0.25	1.03	1.62
208	995	0.05	8.00	0.67	0.25	0.97	1.44
209	370	0.10	10.10	0.91	0.25	1.46	1.61
210	584	0.00	11.20	1.01	0.50	1.31	1.30
211	1263	0.05	20.40	0.85	0.25	1.49	1.76
212	774	0.10	22.50	1.38	0.60	2.67	1.94
213	41	0.25	55.00	16.25	2.00	20.50	1.26
214	381	0.10	7.40	0.59	0.25	0.71	1.19
215	828	0	17.10	1.26	0.50	2.35	1.86
216	505	0.10	18.10	1.89	0.60	3.43	1.82
217	467	0.20	7.80	0.68	0.25	1.20	1.78
218	633	0.10	18.80	1.09	0.50	1.78	1.63
219	310	0.20	9.50	0.83	0.25	1.71	2.05
220	346	0.10	6.90	0.71	0.25	1.30	1.83
221	495	0.10	10.60	0.55	0.25	1.13	2.06
LG (99)	58312	0	27.30	0.45	0.25	0.87	1.93

Composites

The assays were composited to two metre lengths for those assays captured within the mineralized domains, starting at domain boundary. Composites were adjusted across the intersection of the domain.

Table 14-78 to Table 14-80 present the descriptive statistics for the 2 m capped composite values for gold, copper, and silver by domain, respectively, in the SW Zone.

Table 14-78: Descriptive Statistics for Capped Au (g/t) Composite Values by Mineralized Domain – SW Zone

Mineralized Zone	Count	Min	Max	Mean	Median	StDev	CV
199	289	0.003	4.30	0.38	0.23	0.51	1.31
200	390	0.003	5.14	0.44	0.27	0.59	1.36
201	707	0.003	3.54	0.33	0.19	0.50	1.52
202	1531	0.003	14.60	0.49	0.28	0.85	1.74
203	1068	0.003	6.49	0.49	0.26	0.71	1.45
204	181	0.003	2.07	0.21	0.12	0.32	1.51
205	402	0.012	3.83	0.34	0.22	0.41	1.20
206	188	0.005	5.00	0.42	0.16	0.73	1.72
207	159	0.004	3.14	0.41	0.24	0.52	1.25
208	546	0.003	4.02	0.30	0.17	0.44	1.49
209	199	0.003	1.74	0.18	0.11	0.25	1.40
210	319	0.005	2.79	0.25	0.18	0.29	1.13
211	679	0.003	4.23	0.37	0.22	0.52	1.40
212	416	0.003	3.62	0.38	0.23	0.52	1.36
213	23	0.01	2.55	0.47	0.18	0.63	1.33
214	225	0.017	4.29	0.42	0.26	0.51	1.21
215	443	0.003	7.00	0.41	0.22	0.66	1.60
216	272	0.006	5.24	0.40	0.23	0.58	1.43
217	251	0.003	3.02	0.43	0.20	0.64	1.49
218	332	0.003	3.20	0.30	0.17	0.43	1.45
219	177	0.004	2.40	0.26	0.13	0.39	1.50
220	185	0.003	2.50	0.29	0.12	0.44	1.53
221	269	0.003	4.59	0.36	0.16	0.57	1.60
LG (99)	31134	0.001	3.15	0.07	0.04	0.13	1.83

Table 14-79: Descriptive Statistics for Capped Cu (%Cu) Composite Values by Mineralized Domain – SW Zone

Mineralized Zone	Count	Min	Max	Mean	Median	StDev	CV
199	289	0	0.29	0.03	0.01	0.05	1.64
200	390	0	0.52	0.05	0.01	0.10	1.90
201	707	0	0.96	0.06	0.04	0.07	1.19
202	1531	0	0.48	0.03	0.02	0.04	1.38
203	1068	0	1.01	0.07	0.03	0.10	1.42
204	181	0	0.37	0.05	0.03	0.05	1.13
205	402	0.001	0.61	0.07	0.05	0.07	1.07
206	188	0	0.12	0.01	0.00	0.02	1.72
207	159	0	0.05	0.01	0.00	0.01	1.44
208	546	0	0.41	0.04	0.02	0.05	1.29
209	199	0.001	0.40	0.05	0.02	0.06	1.34
210	319	0	0.59	0.06	0.04	0.08	1.27
211	679	0	0.40	0.06	0.03	0.06	1.14
212	416	0.001	0.72	0.05	0.03	0.06	1.22
213	23	0.012	0.04	0.02	0.02	0.01	0.32
214	225	0.001	0.28	0.05	0.03	0.06	1.12
215	443	0	0.49	0.06	0.04	0.08	1.28
216	272	0.001	0.42	0.05	0.02	0.06	1.32
217	251	0	0.13	0.01	0.01	0.02	1.47
218	332	0.001	0.47	0.05	0.04	0.05	1.01
219	177	0	0.11	0.01	0.01	0.02	1.45
220	185	0	0.15	0.02	0.00	0.03	1.78
221	269	0	0.09	0.01	0.00	0.02	1.41
LG (99)	31134	0	0.74	0.02	0.01	0.02	1.28

Table 14-80: Descriptive Statistics for Capped Ag (g/t) Composite Values by Mineralized Domain – SW Zone

Mineralized Zone	Count	Min	Max	Mean	Median	StDev	CV
199	289	0.10	22.00	1.61	0.69	2.57	1.59
200	390	0.10	17.93	1.16	0.55	1.76	1.51
201	707	0.10	15.75	1.18	0.69	1.59	1.34
202	1531	0.10	16.60	0.56	0.25	0.78	1.40
203	1068	0.10	15.06	0.89	0.42	1.36	1.53
204	181	0.20	7.40	0.95	0.60	1.02	1.08
205	402	0.20	11.35	1.18	0.70	1.35	1.14
206	188	0.10	5.00	0.60	0.25	0.82	1.36
207	159	0.20	4.85	0.64	0.25	0.84	1.30
208	546	0.10	5.77	0.67	0.38	0.77	1.13
209	199	0.20	9.65	0.90	0.51	1.16	1.29
210	319	0.20	7.25	1.03	0.63	1.18	1.15
211	679	0.13	12.30	0.85	0.48	1.16	1.36
212	416	0.10	22.50	1.38	0.65	2.33	1.68
213	23	0.25	51.93	15.14	2.06	19.19	1.27
214	225	0.10	3.95	0.63	0.39	0.59	0.94
215	443	0.05	17.10	1.30	0.63	1.94	1.48
216	272	0.20	18.10	1.98	0.68	3.28	1.66
217	251	0.20	7.80	0.67	0.25	0.96	1.44
218	332	0.10	11.50	1.10	0.60	1.41	1.28
219	177	0.18	4.50	0.66	0.25	0.88	1.34
220	185	0.20	4.70	0.67	0.25	0.93	1.39
221	269	0.10	7.70	0.55	0.25	0.94	1.70
LG (99)	31141	0	27.30	0.43	0.25	0.63	1.46

Variography

Spatial analysis was performed on 2 m capped composite gold data within the mineralized domains. The same experimental variograms were employed for copper and silver; and the variogram parameters were assigned to each mineralized domain. Table 14-81 presents the variogram parameters for the SW Zone for gold values by domain.

Table 14-81: Variogram Parameters by Domain –SW Zone

Domain	Nugget	Structures Sill=1.0	Dip (°)	Dip Az. (°)	Pitch (°)	Major (m)	Semi-Major (m)	Minor (m)	Anisotropy
199	0.4	0.4	52	280	106	150	100	20	Spheroidal
		0.2	52	280	106	200	150	40	Spheroidal
200	0.4	0.4	54	280	106	150	100	20	Spheroidal
		0.2	54	280	106	200	150	40	Spheroidal
201	0.4	0.4	68	286	106	150	100	20	Spheroidal
		0.2	68	286	106	200	150	40	Spheroidal
202	0.4	0.4	66	290	108	150	100	20	Spheroidal
		0.2	66	290	108	200	150	40	Spheroidal
203	0.4	0.4	62	290	116	150	100	20	Spheroidal
		0.2	62	290	116	200	150	40	Spheroidal
204	0.4	0.4	64	282	106	150	100	20	Spheroidal
		0.2	64	282	106	200	150	40	Spheroidal
205	0.4	0.4	66	286	106	150	100	20	Spheroidal
		0.2	66	286	106	200	150	40	Spheroidal
206	0.4	0.4	46	288	106	150	100	20	Spheroidal
		0.2	46	288	106	200	150	40	Spheroidal
207	0.4	0.4	48	282	106	150	100	20	Spheroidal
		0.2	48	282	106	200	150	40	Spheroidal
208	0.4	0.4	68	286	106	150	100	20	Spheroidal
		0.2	68	286	106	200	150	40	Spheroidal
209	0.4	0.4	62	282	106	150	100	20	Spheroidal
		0.2	62	282	106	200	150	40	Spheroidal
210	0.4	0.4	66	284	106	150	100	20	Spheroidal
		0.2	66	284	106	200	150	40	Spheroidal
211	0.4	0.4	68	290	112	150	100	20	Spheroidal
		0.2	68	290	112	200	150	40	Spheroidal
212	0.4	0.4	68	288	106	150	100	20	Spheroidal
		0.2	68	288	106	200	150	40	Spheroidal
213	0.4	0.4	64	286	106	150	100	20	Spheroidal
		0.2	64	286	106	200	150	40	Spheroidal
214	0.4	0.4	64	286	106	150	100	20	Spheroidal
		0.2	64	286	106	200	150	40	Spheroidal
215	0.4	0.4	68	286	106	150	100	20	Spheroidal
		0.2	68	286	106	200	150	40	Spheroidal
216	0.4	0.4	70	286	106	150	100	20	Spheroidal
		0.2	70	286	106	200	150	40	Spheroidal
217	0.4	0.4	48	294	106	150	100	20	Spheroidal
		0.2	48	294	106	200	150	40	Spheroidal
218	0.4	0.4	66	286	106	150	100	20	Spheroidal
		0.2	66	286	106	200	150	40	Spheroidal
219	0.4	0.4	50	292	106	150	100	20	Spheroidal
		0.2	50	292	106	200	150	40	Spheroidal

Domain	Nugget	Structures Sill=1.0	Dip (°)	Dip Az. (°)	Pitch (°)	Major (m)	Semi-Major (m)	Minor (m)	Anisotropy
220	0.4	0.4	48	292	106	150	100	20	Spheroidal
		0.2	48	292	106	200	150	40	Spheroidal
221	0.4	0.4	52	292	106	150	100	20	Spheroidal
		0.2	52	292	106	200	150	40	Spheroidal
99(LG)	0.4	0.4	52	292	106	150	100	20	Spheroidal
		0.2	52	292	106	200	150	40	Spheroidal

Note: Az – Azimuth

14.8.3 Block Model

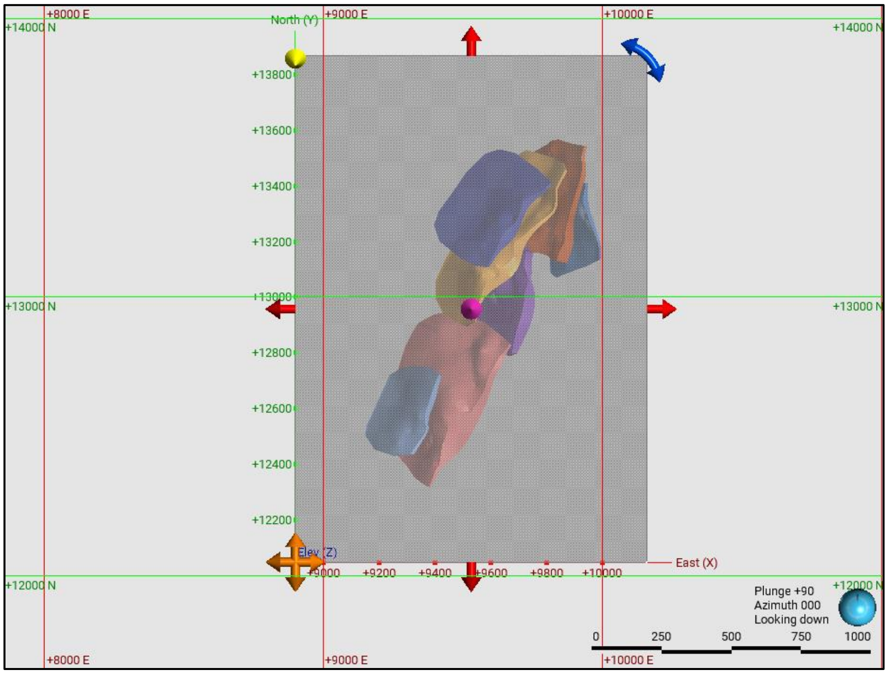
Block Model Parameters

The block model for the SW Zone deposit was set up with a block matrix of 5 m long by 5 m wide by 5 m high and was created using Leapfrog Edge resource software. The block matrix was defined based on current drill hole spacing and on engineering considerations for an open pit operation and is considered suitable this purpose. The block model is in mine grid coordinates and is not rotated. Table 14-82 summarizes the block model parameters and Figure 14-27 presents the block model over the interpreted mineralized domains for the SW Zone.

Table 14-82: Block Model Parameters – SW Zone

Domain	Minimum	Maximum
Easting	8400 mE	10600 mE
Northing	7650 mN	12000 mN
Maximum Elevation	4400 m	5500 m
Rotation Angle	No rotation°	
Block Size (X, Y, Z in metres)	5 x 5 x 5	
Number of blocks in the X direction	440	
Number of blocks in the Y direction	870	
Number of blocks in the Z direction	220	

Figure 14-27: Block Model Extents – SW Zone



Source: AGP (2023)

The block model extents cover the entire SW Zone and is extended on all four sides beyond the interpreted mineralized domains. The block model is a whole block model where blocks are assigned a specific rock type code. Any block centroid within the mineralized domain wireframe was assigned that code.

Block model attributes in the block model includes:

- domain code
- metal grades for gold, copper, silver, and calculated gold-equivalent grades for estimated blocks
- classification
- distance to the nearest composite
- average distance of estimated composites
- number of composites used in estimation of a block
- number of drill holes used in estimation of a block
- pass number
- lithology code
- density

Estimation Parameters and Interpolation Strategy

The metal grades were interpolated in three passes using the 2 m capped composites by domain. The metal grades were interpolated using OK interpolation method. Variogram parameters for each metal was used in each of these passes and used variable orientation aligned to the mineralized domain wireframe. ID^2 and NN interpolations were also run for validation purposes.

Each pass required the same minimum and maximum number of composites for each domain. A maximum of three composites per drill hole was used. Table 14-83 shows estimation parameters for each pass used to estimate metal grades.

Table 14-83: Estimation Parameters – SW Zone

Pass	Min No Composites	Max No Composites	Max No. Composites per Hole	Min. No. of Drill Holes
Pass 1	4	18	3	2
Pass 2	4	18	3	2
Pass 3	3	18	3	1

Due to the sinuous nature of the interpreted mineralized domains, search ellipses were allowed a variable orientation that follow the trend of each domain. Each pass increased the search ellipse where Pass 2 was doubled that of Pass 1 and Pass 3 was approximately double that of Pass 2. Hard boundaries were kept between all domains and blocks within each domain were estimated only be composites within the domain wireframe.

Table 14-84 shows search ellipse parameters for each subdomain used to estimate metal grades.

Table 14-84: Search Ellipse Parameters for Gold, Copper, and Silver– SW Zone

All Domains	Orientation	Major (m)	Semimajor (m)	Minor (m)	Search
Pass 1	Variable Orientation	75	50	10	Ellipsoidal
Pass 2	Variable Orientation	150	100	20	Ellipsoidal
Pass 3	Variable Orientation	200	150	30	Ellipsoidal

Block Model Validation

The block model was validated using the following methods:

- statistical comparison of resource assay and block grade distributions
- visual inspection and comparison of block grades with composite and assay grades
- inspection of swath plots with composites and block grades elevations and northings

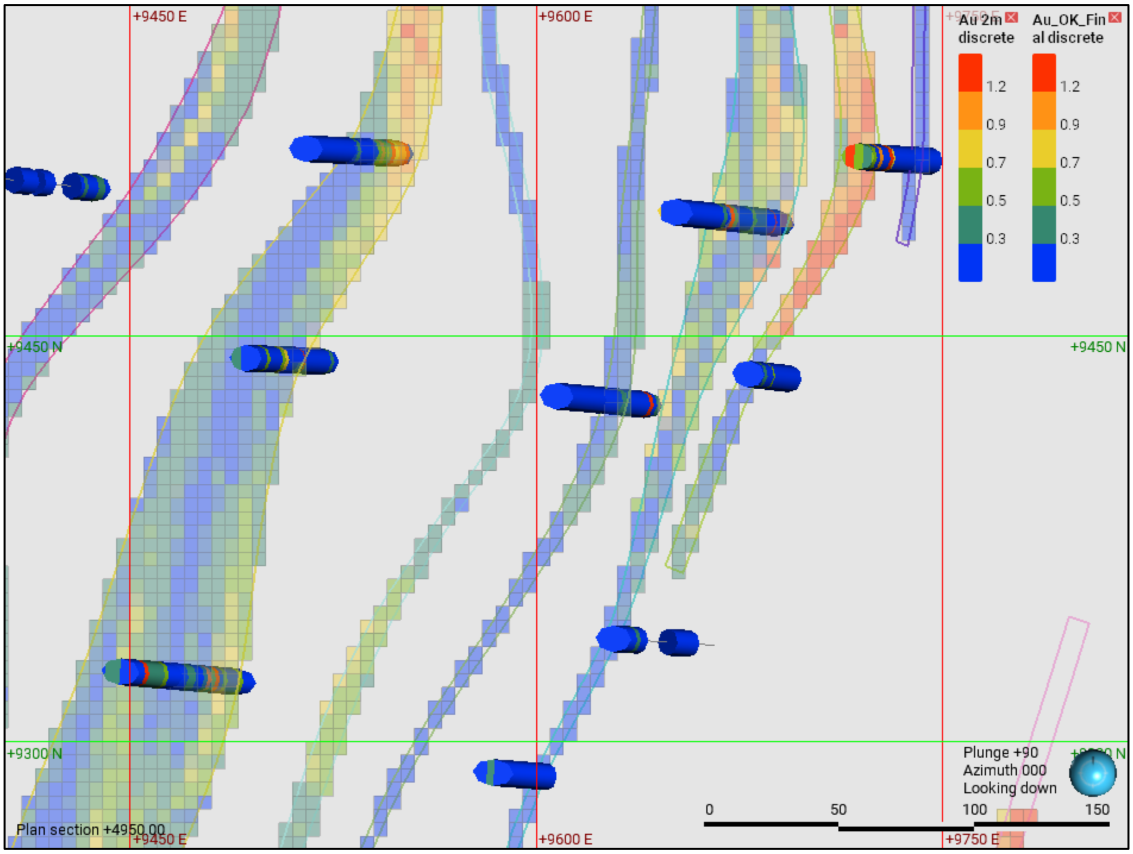
Table 14-85 shows the comparison of mean gold grades by domain for estimated blocks to the ID^2 and NN interpolations and the 2 m composite values. No bias is noted.

Table 14-85: Comparison of Mean Gold Grades (g/t Au) – SW Zone

Domain	OK	ID^2	NN	2 m Comps
199	0.36	0.35	0.39	0.38
200	0.39	0.39	0.42	0.44
201	0.30	0.30	0.30	0.33
202	0.46	0.46	0.48	0.49
203	0.46	0.45	0.46	0.49
204	0.23	0.24	0.19	0.21
205	0.34	0.34	0.36	0.34
206	0.39	0.38	0.34	0.42
207	0.38	0.37	0.41	0.41
208	0.31	0.31	0.30	0.30
209	0.18	0.18	0.17	0.18
210	0.25	0.25	0.25	0.25
211	0.38	0.38	0.39	0.37
212	0.36	0.35	0.39	0.38
213	0.50	0.50	0.63	0.47
214	0.36	0.36	0.35	0.42
215	0.39	0.38	0.32	0.41
216	0.36	0.36	0.36	0.40
217	0.39	0.39	0.39	0.43
218	0.26	0.26	0.30	0.30
219	0.23	0.23	0.21	0.26
220	0.26	0.26	0.25	0.29
221	0.32	0.32	0.35	0.36
99 (LG)	0.07	0.07	0.07	0.07

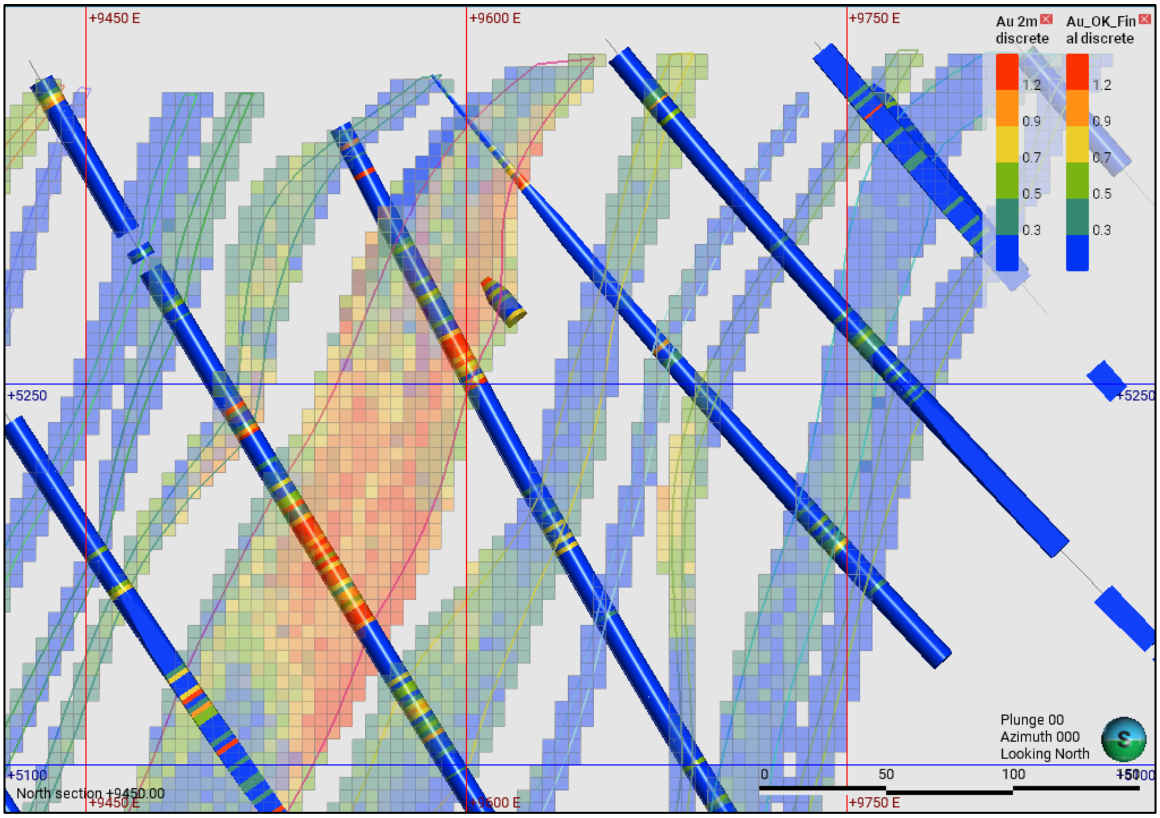
The block model grades, and the composites grades were visually inspected on plan and cross sections. Composite grades honour the block grades well within the mineralized domains. Figure 14-28 and Figure 14-29 present selected plan section and cross section views for the SW Zone (plan view elevation 4,950 m and cross section 9,450 mN), respectively.

Figure 14-28: Plan-section Elevation 4950 m – SW Zone



Source: AGP (2023)

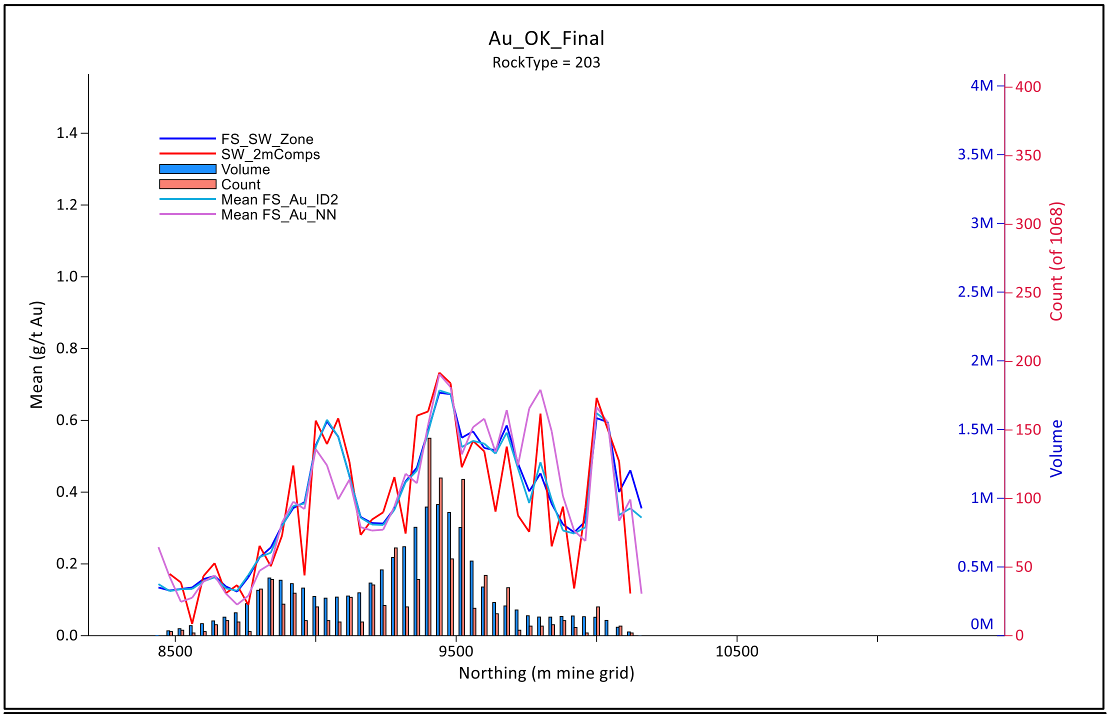
Figure 14-29: Cross-section 9450 mN – SW Zone

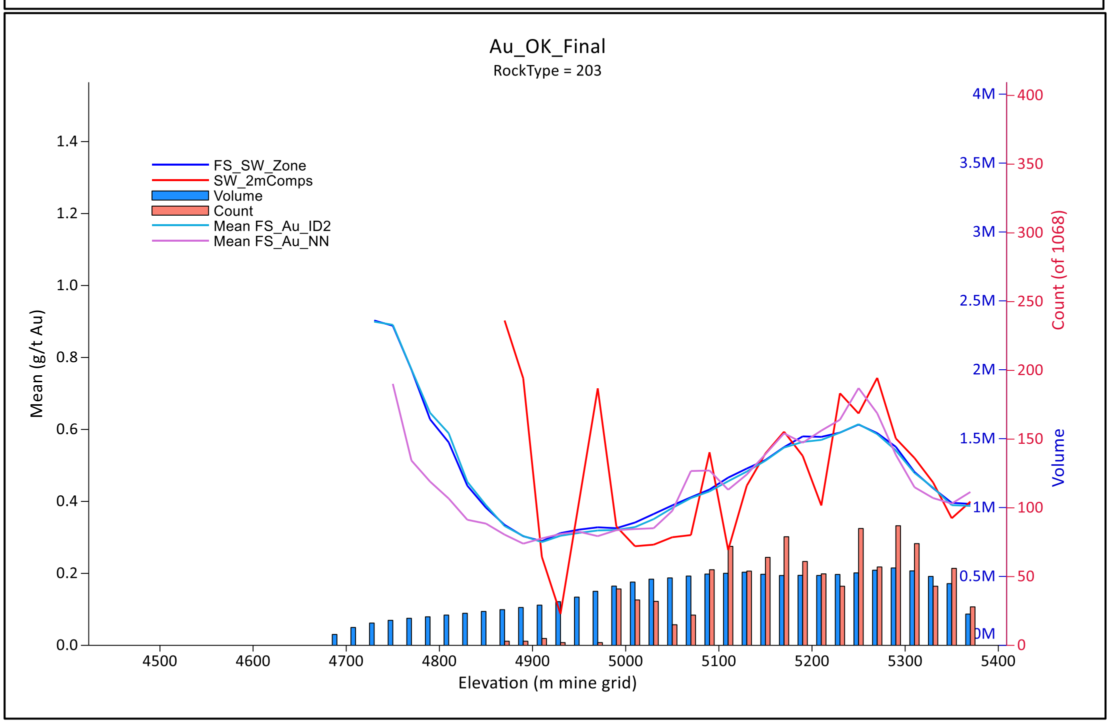


Source: AGP (2023)

Swath plots were reviewed by northing, easting, and elevation. The distribution of gold, copper and silver composite values and interpolated block grades were compared to the OK, ID^2 and NN grades. No issues were found with the distribution of interpolated grades. Figure 14-30 and Figure 14-31 present the swath plots for gold and copper for domain 203 in the SW Zone by northing and elevation, respectively.

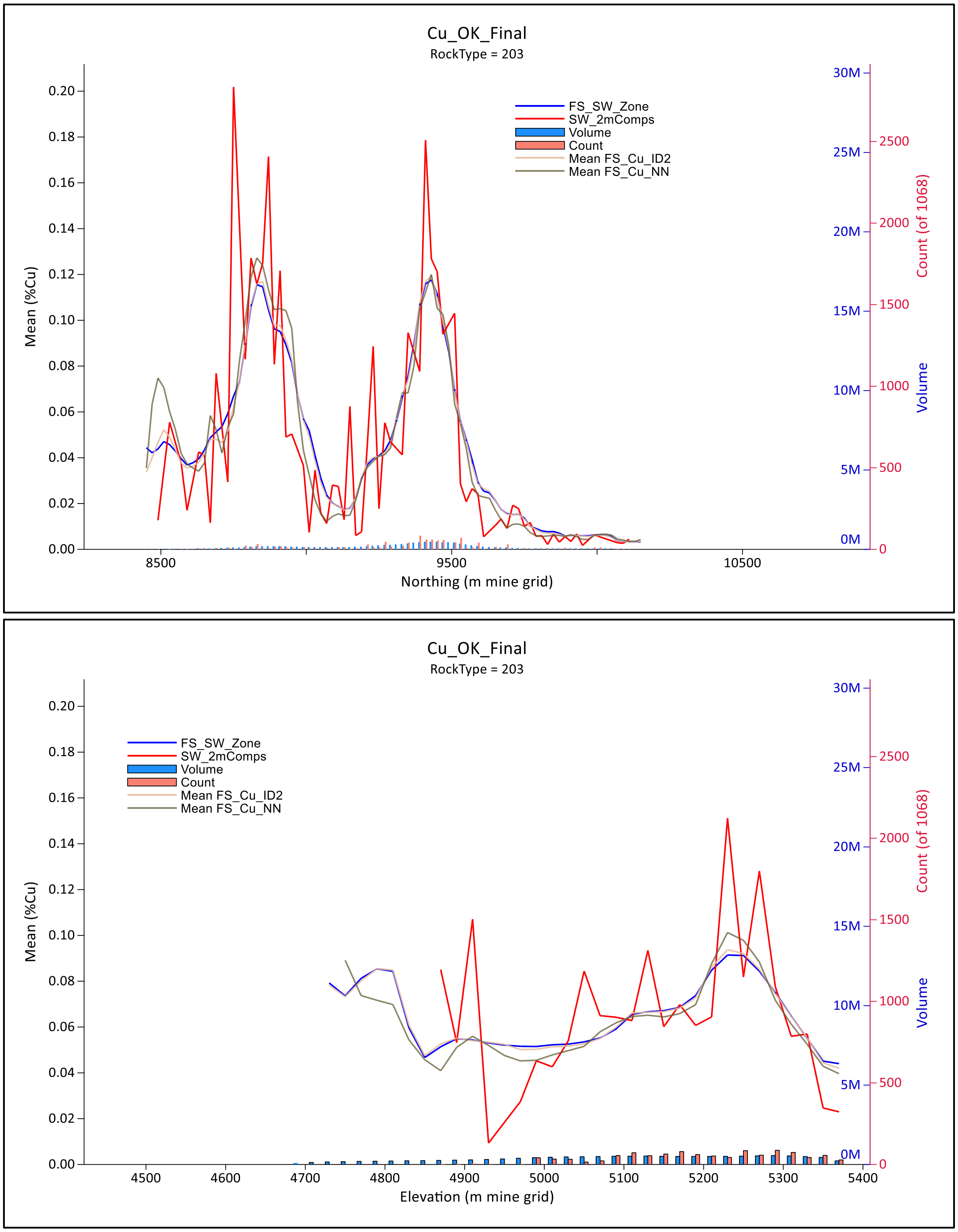
Figure 14-30: Swath Plot for Gold, by Northing and Elevation; Domain 203 – SW Zone





Source: AGP (2023)

Figure 14-31: Swath Plot for Copper, by Northing and Elevation; Domain 203 – SW Zone



Source: AGP (2023)

14.9 Mineral Resources

14.9.1 Mineral Resource Classification

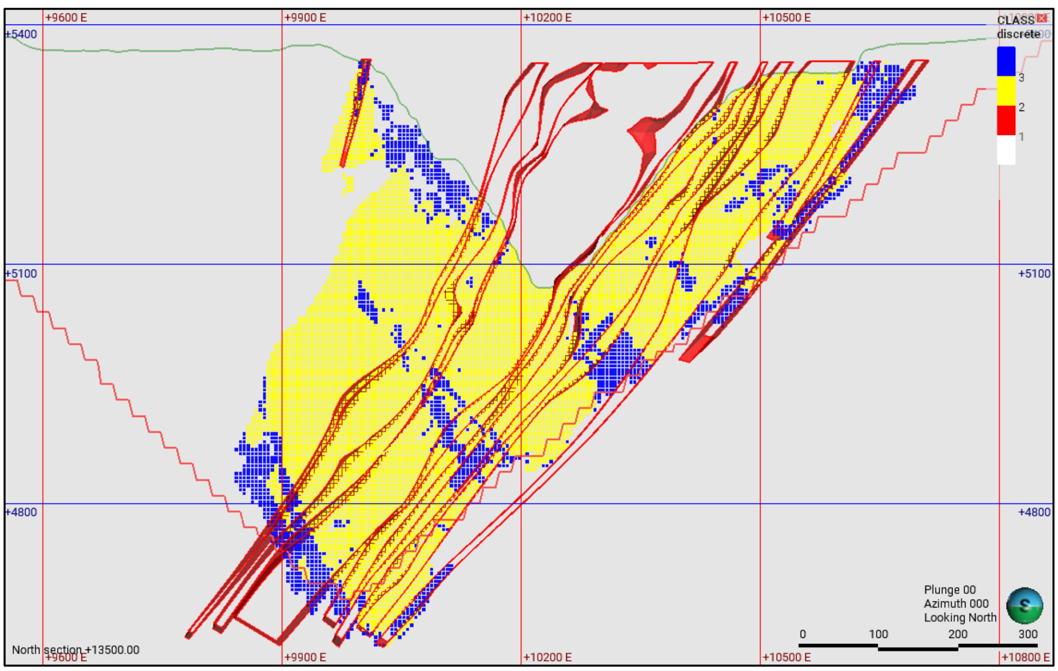
Mineral resources were classified in accordance with definitions provided by CIM (2014) Standards and Definitions. The mineral resources at the Project were classified as Inferred and Indicated mineral resources.

Z87 Zone

For the Z87 Zones, blocks interpolated in the first or second pass with a minimum of two holes, and an average distance for the composites of less than 65 m were initially classified as Indicated resources. Blocks interpolated in the second or third pass with a minimum of two drill holes and an average distance to composites greater than or equal to 65 m and less than 120 m were initially classified as Inferred resources. Any blocks interpolated in pass 3 with an average distance to composites greater than or equal to 120 m were not included in any resource categories and they were assigned a Code 4.

Figure 14-32 shows a representative cross section of classified blocks for the Z87 Zone.

Figure 14-32: Cross-section 13,500 mN; showing class blocks and optimized pit constraint – Z87 Zone



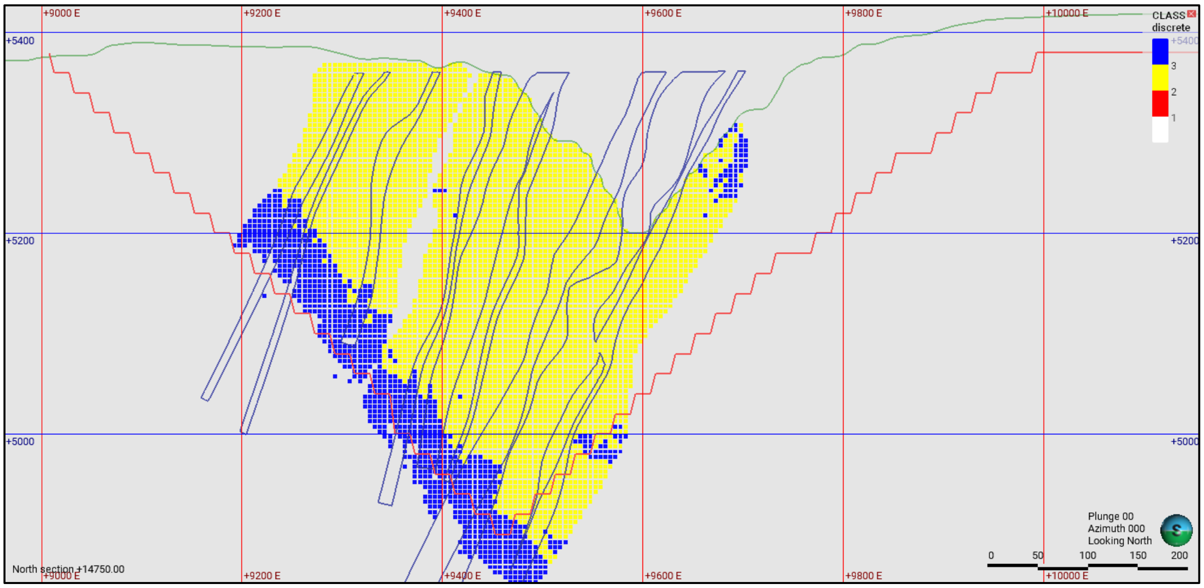
Source: AGP (2023)
Yellow – Indicated, Blue – Inferred

J Zone

For J Zone, blocks interpolated in the first or second pass with a minimum of six composites, minimum of two drill holes, and an average distance to composites of less than 60 m were classified as Indicated resources. Blocks interpolated in the first, second or third pass with a minimum of one sample per drillhole, a minimum of one drillhole, and an average distance to composites less than 120 m were classified as Inferred resources. Any blocks greater than 120 m were not included in any resource categories. The Indicated blocks were coded with the number 2 and the Inferred blocks were coded with the number 3.

Figure 14-33 shows a representative cross section of classified blocks for the J Zone.

Figure 14-33: Cross-section 14750 mN; showing class blocks and optimized pit constraint – J Zone



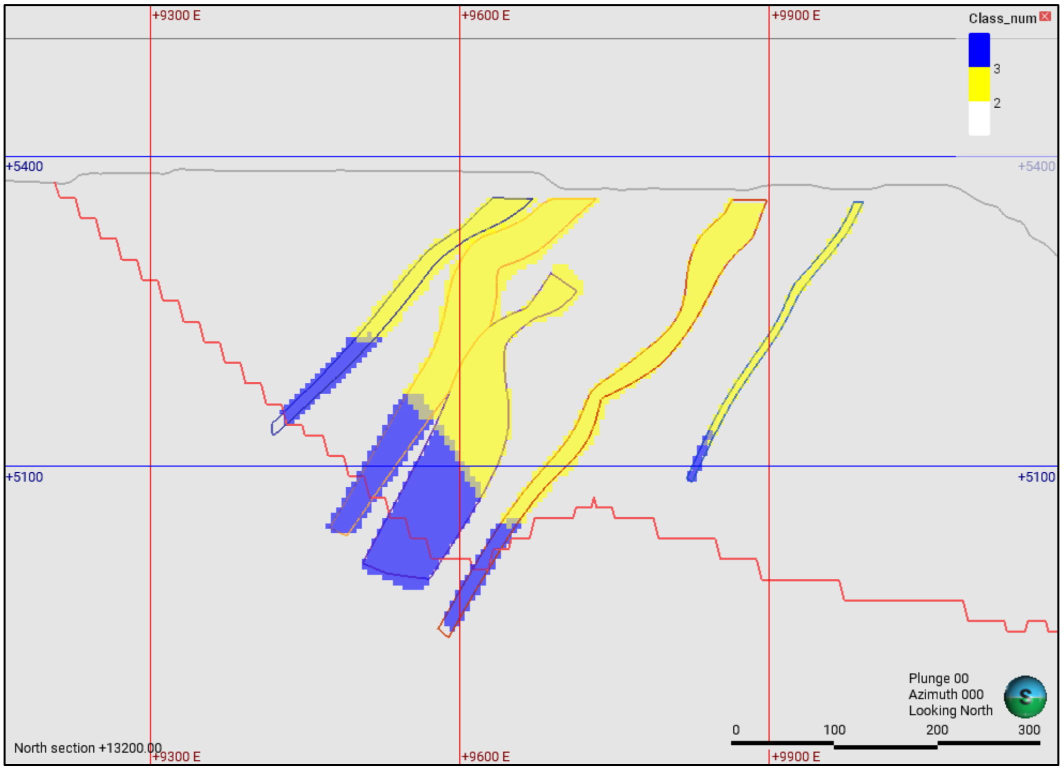
Source: AGP (2023)
Yellow – Indicated, Blue – Inferred

X22 Zone

For the X22 Zone, Blocks Interpolated with a minimum of four composite values, or two drill holes, a nearest distance of nominally 65 m or average distance of 70 m, were initially classified as Indicated mineral resources. Mineralized domains were examined zone-by-zone and consolidated contiguous blocks manually to upgrade or downgrade isolated blocks. Blocks interpolated with a minimum of two drill holes and up to a nearest distance of 120 m were classified as Inferred mineral resources.

Figure 14-34 presents a representative cross section (13,200 mN) showing classed blocks in the main mineralized domains and optimized pit constraint for the X22 Zone.

Figure 14-34: Cross-section 13200 mN; showing class blocks and optimized pit constraint – X22 Zone



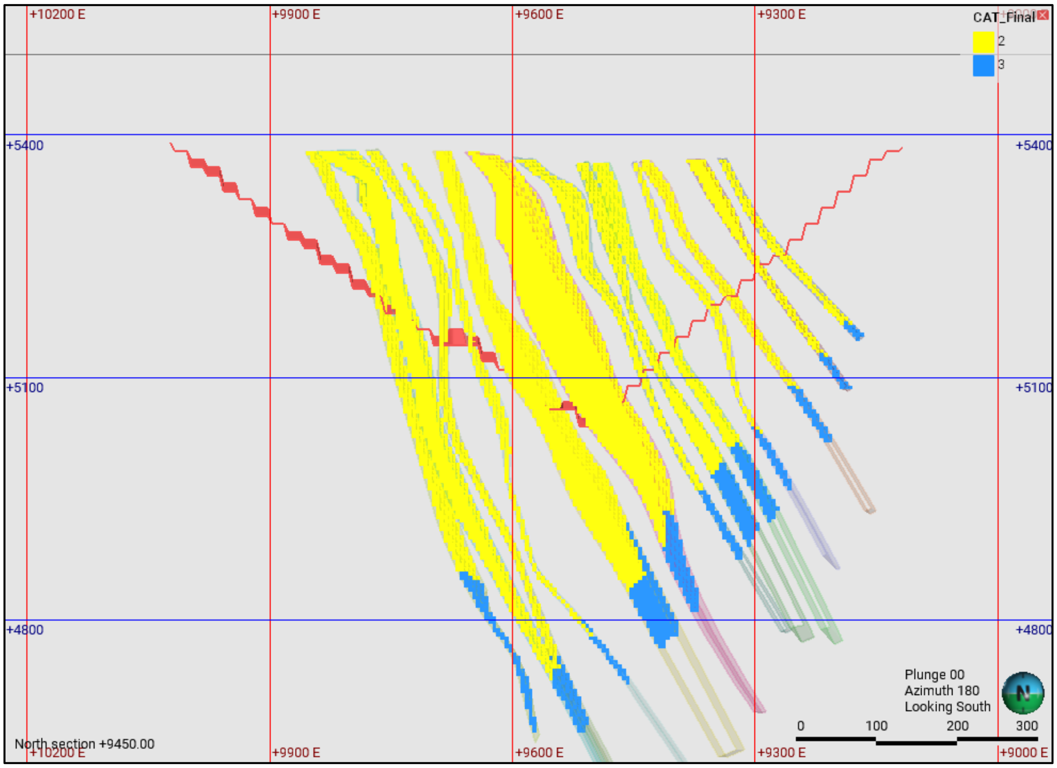
Source: AGP (2023)
Yellow – Indicated, Blue – Inferred

SW Zone

For the SW Zone, Blocks Interpolated with a minimum of four composite values, or two drill holes, a nearest distance of nominally 65 m or average distance of 70 m, were initially classified as Indicated mineral resources. Mineralized domains were examined zone-by-zone and consolidated contiguous blocks manually to upgrade or downgrade isolated blocks. Blocks interpolated with a minimum of two drill holes and up to a nearest distance of 120 m were classified as Inferred mineral resources.

Figure 14-35 presents a representative cross section (9,450 mN) of classed blocks in the main mineralized domains for the SW Zone.

Figure 14-35: Cross-section 9450 mN; showing class blocks and optimized pit constraint – SW Zone



Source: AGP (2023)
Yellow – Indicated, Blue – Inferred

14.9.2 Metal Equivalent

A metal equivalent grade was used to determine cut-off grades for the Project. Metal equivalent grades are used in determining an equivalent value for a block by including the influence of other metal grades in the same block. The principal credit for the Project is gold, therefore, a gold equivalent (AuEQ) was used. The AuEQ grades were calculated based on the capped grades from the OK interpolation for all Zone. The AuEQ grades were calculated for each block after metal grade interpolations were completed using the following equations:

$$Z87\ Zone \qquad AuEq = Au\ grade + (1.5628\ x\ Cu\ grade) + (0.0128\ x\ Ag\ grade)$$

$$J\ Zone \qquad AuEq = Au\ grade + (1.5107\ x\ Cu\ grade) + (0.0119\ x\ Ag\ grade)$$

$$SW\ Zone \qquad AuEq = Au\ grade + (1.6823\ x\ Cu\ grade) + (0.0124\ x\ Ag\ grade)$$

$$X22\ Zone \qquad AuEq = Au\ grade + (1.5628\ x\ Cu\ grade) + (0.0128\ x\ Ag\ grade)$$

Table 14-86 lists the parameters used in the above formulas

Table 14-86: AuEQ Formula Parameters, by Zone

		Metal Recovery (%)			
Metal	**Metal Price (US$)**	**Z87**	**JZ**	**SW**	**X22**
Gold	1,850.00/oz	95.5	93.1	85.7	95.5
Copper	4.25/lb	94.7	89.3	91.5	94.7
Silver	25.00/oz	98.2	88.9	85.6	98.2

14.9.3 Cut-off Grade

For all Zones at the Project, AGP has determined a resource cut-off grade of 0.3 g/t AuEQ to be used for reporting of the mineral resources within constraining shells for the material amenable to open pit extraction. A resource cut-off grade of 0.9 g/t AuEQ for material that may be amenable to underground extraction was applied for contiguous blocks below the constraining shells, captured within 0.9 g/t AuEQ grade shells. The cut-off grades are based on the parameters defined below.

14.9.4 Reasonable Prospects for Eventual Economic Extraction

The block models were exported to develop an optimized constraining shell to satisfy the Reasonable Prospects for Eventual Economic Extraction for the Project. The three northern zones were merged into a single block model and, with the SW Zone model, were exported in ASCII format and imported into Hexagon MineSight®. Table 14-87 shows the economic assumptions made to constrain the reported mineral resources.

Table 14-87: Parameters for Constraining Shells, by Zone

Parameter	Unit	Z87 Pit Area 2	X22 Pit Area 4	J Pit Area 1	SW Pit Area 3
Metal Prices					
Gold	US$/oz	1,850.00	1,850.00	1,850.00	1,850.00
Copper	US$ /lb	4.25	4.25	4.25	4.25
Silver	US$ /oz	25.00	25.00	25.00	25.00
Exchange	Ratio CAD/USD	1.30:1	1.30:1	1.30:1	1.30:1
Metal Recoveries (overall)					
Gold	%	96	94	93	84
Copper	%	95	94	91	87
Silver	%	98	98	90	92
Costs					
Mining Rate – OP ore	tpd	35,000	35,000	35,000	35,000
Mining Cost – OP waste	C$/t total	1.99	2.15	2.15	2.01
Mining Cost – OP ore	C$ /t total	2.10	2.29	2.29	2.37
Incremental Cost waste per 10 m below 5360 m	C$ /10 m bench	0.04	0.04	0.04	0.04
Incremental Cost ore per 10 m below 5360 m	C$ /10 m bench	0.03	0.04	0.04	0.03
Processing Cost	C$ /t mill feed	7.29	7.29	7.29	7.29
G&A Cost	C$ /t mill feed	1.76	1.76	1.76	1.76
OP Slope Angles					
West	degrees	44	46	46	43
North	degrees	48-43	46	49	47
East	degrees	39	40	43	41
South	degrees	48	46	47	45

OP – Open Pit; G&A – General and Administration

For reasonable prospects for eventual economic extraction for mineral resources amenable to underground extraction, consolidated wireframes were created below the constraining shells. The consolidated wireframes were developed using net value per tonne of C$ 60/t to capture contiguous blocks amenable to underground resources. The net value per tonne has processing and G&A costs of C$ 9.05/t removed so is reflective of approximate underground mining costs.

14.10 Mineral Resource Statement

The mineral resources for the Project deposit amenable to open pit extraction at a 0.3 g/t AuEQ cut-off grade are: Indicated Resource of 506.2 Mt at 0.57 g/t Au, 0.07 %Cu, 1.09 g/t Ag and 0.68 g/t AuEQ; and an Inferred Resource of 76.5 Mt at 0.53 g/t Au, 0.06 %Cu, 1.12 g/t Ag and 0.65 g/t AuEQ. Table 14-88 presents the Mineral Resources amenable to open pit extraction. The effective date of the Mineral Resources is 2 October 2023.

Table 14-88: Open Pit Mineral Resources for the Troilus Project at a 0.3 g/t AuEQ Cut-off Grade – All Zones

		Grade				Contained Metal			
Class	Tonnes (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	AuEQ (g/t)	Au (Moz)	Cu (Mlb)	Ag (g/t)	AuEQ (Moz)
Z87									
Indicated	197.1	0.67	0.07	1.21	0.80	4.21	320.69	7.67	5.04
Inferred	37.1	0.59	0.06	1.11	0.70	0.71	50.17	1.33	0.84
JZ									
Indicated	151.9	0.50	0.06	0.96	0.61	2.45	215.71	4.71	2.98
Inferred	24.2	0.46	0.07	0.94	0.57	0.35	35.37	0.73	0.44
X22									
Indicated	59.2	0.51	0.06	1.24	0.62	0.98	79.34	2.35	0.19
Inferred	13.6	0.53	0.07	1.48	0.67	0.23	21.76	0.65	0.29
SW									
Indicated	98.0	0.50	0.05	0.94	0.60	1.59	109.91	2.94	1.89
Inferred	1.6	0.37	0.04	0.96	0.45	0.02	1.36	0.05	0.02
TOTALS – ALL ZONES									
Indicated	506.2	0.57	0.07	1.09	0.68	9.23	725.66	17.67	11.11
Inferred	76.5	0.53	0.06	1.12	0.65	1.31	108.66	2.75	1.59

Notes:

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Summation errors may occur due to rounding.

Open pit mineral resources are reported within optimized constraining shells.

Open pit cut-off grade is 0.3 g/t AuEQ.

AuEQ equivalents were calculated as follows:

Z87 Zone AuEQ = Au grade + 1.5628 * Cu grade + 0.0128 * Ag grade
J4/J5 Zone AuEQ = Au grade + 1.5107 * Cu grade + 0.0119 * Ag grade
SW Zone AuEQ = Au grade + 1.6823 * Cu grade + 0.0124 * Ag grade
X22 Zone AuEQ = Au grade + 1.5628 * Cu grade + 0.0128* Ag grade

Metal prices for the AuEQ formulas are: US$ 1,850/ oz Au; $4.25/lb Cu, and $25.00/ oz Ag; with an exchange rate of US$1.00:CAD$1.30

Metal recoveries for the AuEQ formulas are:

Z87 Zone 95.5% for Au recovery, 94.7% for Cu recovery and 98.2% for Ag recovery
J Zone 93.1% for Au recovery, 89.3% for Cu recovery and 88.9% for Ag recovery
SW Zone 85.7% for Au recovery, 91.5% for Cu recovery and 85.6% for Ag recovery
X22 Zone 95.5% for Au recovery, 94.7% for Cu recovery and 98.2% for Ag recovery

Capping of grades varied between 2.30 g/t Au and 21.00 g/t Au; between 0.06% cu and 4.36 %Cu, and between 3.20 g/t Ag and 55.00 g/t Ag; on raw assays.

The density (excluding overburden and fill) varies between 2.64 g/cm^3 and 2.93 g/cm^3 depending on lithology for each zone.

The mineral resources for the Project deposit amenable to underground extraction at a 0.9 g/t AuEQ cut-off grade are: an Indicated Resource of 2.1 Mt at 1.35 g/t Au, 0.09 %Cu, 1.90 g/t Ag and 1.51 g/t AuEQ; and an Inferred Resource of 4.0 Mt at 1.36 g/t Au, 0.08 %Cu, 8.21 g/t Ag and 1.58 g/t AuEQ. Table 14-89 presents the Mineral Resources amenable to underground extraction.

Table 14-89: Underground Mineral Resources for Troilus Project at 0.9 g/t AuEQ Cut-off Grade – all zones

Class	Tonnes (Mt)	Grade				Contained Metal			
		Au (g/t)	Cu (%)	Ag (g/t)	AuEQ (g/t)	Au (Moz)	Cu (Mlb)	Ag (g/t)	AuEQ (Moz)
Z87									
Indicated	0.5	1.59	0.15	0.54	1.83	0.02	1.55	0.01	0.03
Inferred	1.1	1.99	0.12	0.46	2.19	0.07	2.96	0.02	0.08
JZ									
Indicated	0.2	1.21	0.07	1.46	1.33	0.01	0.29	0.01	0.01
Inferred	1.0	1.25	0.05	0.99	1.34	0.04	1.13	0.03	0.04
X22									
-none-									
-none-									
SW									
Indicated	1.4	1.28	0.07	2.44	1.42	0.06	2.00	0.11	0.06
Inferred	1.9	1.05	0.06	16.62	1.37	0.06	2.66	1.01	0.08
TOTALS – ALL ZONES									
Indicated	2.1	1.35	0.09	1.90	1.51	0.09	3.84	0.13	0.10
Inferred	4.0	1.36	0.08	8.21	1.58	0.18	6.75	1.06	0.20

Notes:
Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
Summation errors may occur due to rounding.
Underground resources reported in 0.9 g/t AuEQ grade shells
Underground cut-off grade is 0.9 g/t AuEQ.
AuEQ equivalents were calculated as follows:
Z87 Zone AuEQ = Au grade + 1.5628 * Cu grade + 0.0128 * Ag grade
J4/J5 Zone AuEQ = Au grade + 1.5107 * Cu grade + 0.0119 * Ag grade
SW Zone AuEQ = Au grade + 1.6823 * Cu grade + 0.0124 * Ag grade
X22 Zone AuEQ = Au grade + 1.5628 * Cu grade + 0.0128* Ag grade
Metal prices for the AuEQ formulas are: US$ 1,850/ oz Au; $4.25/lb Cu, and $25.00/ oz Ag; with an exchange rate of US$1.00: CAD$1.30.
Metal recoveries for the AuEQ formulas are:
Z87 Zone 95.5% for Au recovery, 94.7% for Cu recovery and 98.2% for Ag recovery
J Zone 93.1% for Au recovery, 89.3% for Cu recovery and 88.9% for Ag recovery
SW Zone 85.7% for Au recovery, 91.5% for Cu recovery and 85.6% for Ag recovery
X22 Zone 95.5% for Au recovery, 94.7% for Cu recovery and 98.2% for Ag recovery
Capping of grades varied between 2.30 g/t Au and 21.00 g/t Au; between 0.06% cu and 4.36 %Cu, and between 3.20 g/t Ag and 55.00 g/t Ag; on raw assays.
The density (excluding overburden and fill) varies between 2.64 g/cm^3 and 2.93 g/cm^3 depending on lithology for each zone.

14.11 Factors That May Affect Mineral Resource Estimate

- metal price and exchange rate assumptions
- changes to the assumptions used to generate the gold equivalent grade cut-off grade
- changes in local interpretations of mineralization geometry and continuity of mineralized zones
- changes to geological and mineralization shape and geological and grade continuity assumptions

- density and domain assignments
- changes to geotechnical, mining, and metallurgical recovery assumptions
- change to the input and design parameter assumptions that pertain to the conceptual pit and stope designs constraining the mineral resources
- assumptions and ability to permit and operate the Project
- assumptions and continued ability to access the site, retain mineral and surface rights titles, maintain environment and other regulatory permits, and maintain the social license to operate

It has been noted in the interpretation of the mineralized domains, mainly near surface (new hanging wall domains) in the Z87 Zone, that some of the pre-2018 drill holes contained intervals that were not sampled for silver. Within the mineralized domains, these intervals are assigned half the detection limit, but may, however, underestimate the silver grades in these locations. Additionally, there are some pre-2018 drill holes that were left unsampled near surface that now intersect interpreted mineralized domains, noted in the X22 Zone. It is recommended, where possible, that the twinning of these pre-2018 drill holes be completed to fill in these intervals with supporting data.

14.12 Comparison With Previous Estimate

14.12.1 Mineral Resource Comparison – Z87 Zone

In the Indicated category, the new resource model shows a tonnage improvement of 112.5 Mt. The tonnage improvement was sufficiently large to offset the reduction in grade and shows an improvement of 2.06 Moz in contained gold ounces. The changes are attributed to a change in the interpreted wireframes based on most recent drilling information. Additionally, the interpolation within a low-grade halo may have contributed in part to the additional tonnage.

The consolidation of three northern deposits of the Project, the Z87, J and X22 Zones, allowed the optimized constraining shell to encapsulate the three deposits. This is an additional contributing factor to the increased tonnes in the Mineral Resources within the pit constraint.

Table 14-90 shows the comparison of Mineral Resources for gold between the October 2023 and the previous July 2020 Mineral Resources.

Table 14-90: Comparison of October 2023 vs July 2020 Mineral Resources (Gold) – Z87 Zone

	AGP (Oct 2023) within 2023 optimized shell ≥ 0.3 g/t AuEQ			AGP (Jul 2020) within 2020 optimized shell ≥ 0.3 g/t AuEQ			Differences		
Class	Tonnes (Mt)	Au (g/t)	Cont'd Au (Moz)	Tonnes (Mt)	Au (g/t)	Cont'd Au (Moz)	Tonnes (Mt)	Au Diff (g/t)	Cont'd Au (Moz)
Indicated	197.1	0.67	4.21	84.6	0.79	2.15	112.5	-0.12	2.06
Inferred	37.1	0.59	0.71	32.7	0.6	0.63	4.4	-0.01	0.08

14.12.2 Mineral Resource Comparison – J Zone

The new mineral resource for the J Zone shows an overall tonnage increase in the Indicated category of approximately 72.3 Mt, with a decrease of approximately 21.7 Mt in the Inferred category. The gold grades show a decrease due to the re-interpretation of mineralized domains based on the most recent drilling information; and on the incorporation of the low-grade halo surrounding the J Zone. The contained gold ounces have increased in the Indicated category which has been attributed improved sample support and changes in the interpreted wireframe domains.

The consolidation of three northern deposits of the Project, the Z87, J and X22 Zones, allowed the optimized constraining shell to encapsulate the three deposits. This is an additional contributing factor to the increased tonnes in the Mineral Resources within the pit constraint.

Table 14-91 shows the comparison of Mineral Resources for gold between the October 2023 and the previous July 2020 Mineral Resources.

Table 14-91: Comparison of October 2023 vs July 2020 Mineral Resources (Gold) – J Zone

	AGP (Oct 2023) within 2023 optimized shell ≥ 0.3 g/t AuEQ			AGP (Jul 2020) within 2020 optimized shell ≥ 0.3 g/t AuEQ			Differences		
Class	Tonnes (Mt)	Au (g/t)	Cont'd Au (Moz)	Tonnes (Mt)	Au (g/t)	Cont'd Au (Moz)	Tonnes (Mt)	Au Diff (g/t)	Cont'd Au (Moz)
Indicated	151.9	0.50	2.45	79.6	0.57	1.47	72.3	-0.07	0.98
Inferred	24.2	0.46	0.35	45.9	0.55	0.82	-21.7	-0.09	-0.47

14.12.3 Mineral Resource Comparison – SW Zone

There has been a large increase in mineral resources primarily due to the additional information from drill programs completed at the SW Zone since the last reported mineral resources in July 2020. This additional drilling has expanded the interpreted mineralized domains along strike and at depth of the deposit. Overall gold grade has decreased due to the new interpretation of mineralized domains, and in the incorporation of a low-grade halo surrounding mineralized domains.

Table 14-92 shows the comparison of Mineral Resources for gold between the October 2023 and the previous July 2020 Mineral Resources

Table 14-92: Comparison of October 2023 vs July 2020 Mineral Resources (Gold) – SW Zone

	AGP (Oct 2023) within 2023 optimized shell ≥ 0.3 g/t AuEQ			AGP (Jul 2020) within 2020 optimized shell ≥ 0.3 g/t AuEQ			Differences		
Class	Tonnes (Mt)	Au (g/t)	Cont'd Au (Moz)	Tonnes (Mt)	Au (g/t)	Cont'd Au (Moz)	Tonnes (Mt)	Au Diff (g/t)	Cont'd Au (Moz)
Indicated	98.0	0.50	1.59	-	-	-	98.0	0.50	1.59
Inferred	1.6	0.37	0.02	22.6	0.70	0.51	-21.0	-0.33	-0.49

15 MINERAL RESERVE ESTIMATES

15.1 Summary

The Project is planned to be an open pit operation using conventional mining equipment. All work is based on the mine plans generated by AGP.

Costs are based on first principles build-up of operating and capital costs for the life of the project with current vendor quotations for consumables and maintenance. Mining capital costs were based on vendor submissions.

The Mineral Reserves for the Project are based on the conversion of the Measured and Indicated Mineral Resources in the current mine plan within the J, 87, X22 and SW open pits. No Measured Mineral Resources are contained in these deposits, so there will be no Proven Reserves. Indicated Mineral Resources are converted directly to Probable Reserves.

The Mineral Resources in the underground areas below the pits are also not considered in the Mineral Reserves at this time. The total Mineral Reserves for the Project are shown in Table 15-1. Some variation may exist due to rounding.

Table 15-1: Proven and Probable Reserves – January 15, 2024

Reserve Class	Tonnage (Mt)	Grades Au (g/t)	Cu (%)	Ag (g/t)	AuEq (g/t)	CuEq (%)	Contained Metal Au (Moz)	Cu (Mlb)	Ag (Moz)	AuEq (Moz)	CuEq (Blb)
Proven	0	0.00	0.000	0.00	0.00	0.00	0.00	0	0.00	0.00	0.00
Probable	380	0.49	0.058	1.00	0.59	0.39	6.02	484	12.15	7.26	3.24
Proven and Probable	**380**	**0.49**	**0.058**	**1.00**	**0.59**	**0.39**	**6.02**	**484**	**12.15**	**7.26**	**3.24**

Note: This mineral reserve estimate has an effective date of January 15, 2024, and is based on the mineral resource estimate dated October 2, 2023, for Troilus Gold by AGP Mining Consultants Inc. The mineral reserve estimate was completed under the supervision of Willie Hamilton, P.Eng. of AGP, who is a Qualified Person as defined under NI 43-101. Mineral Reserves are stated within the final pit designs based on a US$1,550/oz gold price, US$20.00/oz silver price and US$3.50/lb copper price. An NSR cut-off of C$9.96/t was used to define reserves. The life-of-mine mining cost averaged C$3.99/t mined, preliminary processing costs were C$8.02/t ore and G&A was C$1.94/t ore placed. The metallurgical recoveries were varied according to gold head grade and concentrate grades. 87 pit recoveries for equivalent grades were 95.5%, 94.7% and 98.2% for gold, copper, and silver respectively. J pit recoveries for equivalent grades were 93.1%, 89.3% and 88.9% for gold, copper, and silver respectively. X22 pit recoveries for equivalent grades were 95.5%, 94.7% and 98.2% for gold, copper, and silver respectively. SW pit recoveries for equivalent grades were 85.7%, 91.5% and 85.6% for gold, copper, and silver respectively. The formulas used to calculate equivalent values are as follows, for 87 Pit AuEq = Au + 1.5361*Cu +0.0133 *Ag, for J Pit AuEq = Au + 1.4849*Cu +0.0123 *Ag, for SW Pit AuEq = Au + 1.6535*Cu +0.0129 *Ag, for X22 Pit AuEq = Au + 1.5361*Cu +0.0133 *Ag.

The initial step for reserve conversion was to develop a Net Smelter Return (NSR) block unit value for Measured and Indicated open pit resource blocks. The NSR calculation was based on the parameters in Table 15-2 and is applicable for a copper concentrate. Economic pit shells were then generated using the NSR block values, pit sector slope criteria, and associated mining, process, and G&A costs. Detailed open pit phases designs and mine production schedules were created and costed to support the financial evaluation of the project.

15.2 Pit Slopes

Based on the available geotechnical and hydrogeological data, the pit designs were based on open pit slope design parameters for the four pits as summarized in Table 15-2. The bedrock inter-ramp angles ranging from 45° to 57°. 20 m high double benches are likely achievable in all bedrock sectors, with recommended catch bench widths ranging from 7.5 m to 14.65 m. The slope design criteria assume that controlled blasting will be implemented. Scaling bench faces and cleaning accumulated material from bench toes is recommended. Overall slope stability will also be very sensitive to water pressure past certain depths. Pro-active depressurization with horizontal drains will be required in some areas to implement the slope design. This is particularly true for the west walls. The Inmet experience in mining 87 and J pits between 1996 and 2009 provides an excellent source of information to develop evidence-based pit slope designs.

Table 15-2: Pit Design Slope Criteria

Pit	Slope	Start Azimuth (degrees)	Material Type	IRA (degrees)	BFA (degrees)	Height Between Berms (m)	Catch Bench Width (m)
J	All slopes		OB, fill	21.8	55	10	18.00
	East	357	Bedrock	47	68	20	10.60
	South	107	Bedrock	55	85	20	12.25
	West	237	Bedrock	52	85	20	13.88
	North	282	Bedrock	57	85	20	11.45
87	All slopes		OB, fill	21.8	55	10	18.00
	East	52	Bedrock	45	65	20	10.70
	South	147	Bedrock	54	85	20	12.65
	West	227	Bedrock	52	85	20	13.85
	Northwest	287	Bedrock	57	85	20	11.15
	Northeast	352	Bedrock	51	85	20	14.65
X22	All slopes		OB, fill	21.8	55	10	18.00
	Northwest	267	Bedrock	55	85	20	12.25
	Southeast	47	Bedrock	47	68	20	10.60
	South & West	177	Bedrock	55	85	20	12.25
SW	All slopes		OB, fill	21.8	55	10	18.00
	East	52	Bedrock	50	68	20	8.50
	South	147	Bedrock	54	70	20	7.50
	West	237	Bedrock	56.5	74	20	7.50
	North	347	Bedrock	52	68	20	7.50

Note: 10 m bench heights during mining

While the past operational experiences suggest that pit dewatering should result in generally depressurized slopes, this should be verified by. Vibrating wire piezometers should be installed to capture drawdown pore pressure information to help evaluate the effectiveness of drains on slope performance. Slope movement monitoring data will be required to calibrate numerical models that may be required for detailed design updates during the life of mine.

Ground support will need to be designed by the site geotechnical engineer during operation based on actual ground conditions. Inmet installed rock bolts for face support, and similar support has been assumed and costed in this study.

15.3 Economic Pit Shell Development

The final pit designs are based on pit shells using the Lerchs–Grossmann (LG) procedure in MinePlan software. The parameters for the pit shells are shown in Table 15-3. The mining cost estimates are based on the use of 227 tonne trucks using an approximate waste dump configuration to determine incremental hauls for mill feed and waste. Copper, gold, and silver are the primary elements used in the revenue calculations.

The smelting terms and recovery assumptions are based on creating a specific copper bulk concentrate for each pit. The target copper concentrate grades for J, 87, X22 and SW pits in this study were 15%, 18%, 15% and 16% respectively when the copper feed grade was less than or equal to 0.15%. When the copper feed grade was above 0.15%, the target copper concentrate grades for J, 87, X22 and SW pits in this study were 18%, 22%, 18% and 20%, respectively.

Table 15-3: Economic Pit Shell Parameters (US Dollars unless otherwise noted)

Description	Units	Value		
Exchange Rates				
CAD	US$ =	1.30		
Metal Prices		**Copper**	**Gold**	**Silver**
Price	$/oz	3.50	1550.00	20.00
Royalty	%	1.0%	1.0%	1.0%
Dore				
Payable	%		99%	99.0%
Selling Cost	$/oz		12.00	1.00
Smelting, Refining, Transportation Terms				
Payable	%	96.6%	98.0%	97.0%
Minimum Deduction	unit, g/dmt	1.1	2	20
Participation (on profits)	%	100%	100%	100%
Smelting Charge	$/dmt	99.00		
Refining	$/oz, $/lb	0.099	5.00	0.50
Concentrate Moisture	%	8%		
Transit Losses	%	0.5%	0	0
Concentrate Trucking Cost	$/wmt	92.27		
Concentrate Port Cost	$/wmt	20.00		
Concentrate Shipping Cost	$/wmt	65.00		
Metallurgical Information				
		Copper	**Gold**	**Silver**
Tails solid assay (TG)		%	g/t	g/t
J zone		0.0050	0.0320	0.1030
SW zone		0.0045	0.0630	0.1285
87 zone		0.0050	0.0290	0.0275
X22 zone		0.0050	0.0320	0.1030
Gravity recovery (GR)		Copper	Gold	Silver
J zone	%	0	33.18	33.18
SW zone	%	0	22.48	22.48
87 zone	%	0	36.33	36.33
X22 zone	%	0	30.66	30.66
Copper Concentrate Grade		Cu Feed Grade	Cu Concentrate Grade	
		Cutover value	<= cutover	> cutover
J zone	%	0.15	15	18
SW zone	%	0.15	16	20
87 zone	%	0.15	18	22
X22 zone	%	0.15	15	18

Losses for Gravity circuit	%	0.025		
Power Cost				
Cost of power	C$/Kwhr	0.035		
Fuel Cost				
Diesel Fuel Cost to site	C$/ l	1.66		
Mining Cost *				
Base Rate – 5360 m Elevation		**J and X22 Zones**	**87 Zone**	**SW**
Waste	C$/t moved	2.15	1.99	2.01
Mill Feed	C$/t moved	2.29	2.10	2.37
Incremental Rate - below 5360 m				
Waste	C$/t moved/ 10 m bench	0.039	0.041	0.036
Mill Feed	C$/t moved/ 10 m bench	0.036	0.029	0.028
Processing and G&A				
Processing Cost	C$/t mill feed	8.02		
G&A Cost	C$/t mill feed	1.94		
Process + G&A	C$/t mill feed	9.96		

* mining costs based on using 227 t haul trucks

15.4 Cut-off

For the statement of open pit reserves for the Project, an NSR value per tonne of C$9.96/t was used as the mill feed cut-off. NSR calculations are inclusive of all revenues and royalties for the copper concentrate. Revenues are based on contributions of gold, copper, and silver metals. The cut-off of C$9.96/t was used to flag initial feed and waste blocks prior to dilution and represents the preliminary process and site G&A costs.

No underground reserves are stated in this report but remain an opportunity for future development.

15.5 Dilution and Mining Recovery

Both the north pit and SW resource models were provided in a whole block format. Whole block models means that for any given block, it is routed as either mill feed or waste. The block size within each of the models was 5 m by 5 m in plan, and 5 m high. The approach used to apply external dilution in an appropriate manner is described in the following sub-sections.

Dilutions skins are considered around a mill feed block if the mill feed block is in contact with neighbouring waste blocks. The first step in this method was to define mill feed and waste blocks. An NSR cut-off value of C$9.96/t was used to define mill feed blocks where only Measured and Indicated blocks would be considered. In these models, there were no Measured blocks. This NSR cut-off value represents the marginal cut-off as it includes the sum of process costs and G&A costs. The second step includes determining the number of waste neighbour contacts for each mill feed block. For each waste neighbour, a dilution tonnage and grade are applied to the mill feed block based on the specified dilution skin thickness. The third step is to reduce the tonnage in the neighbouring waste blocks to remove the diluting material.

The dilution material is added to the parent mill feed block and removed accordingly in neighbouring waste blocks to achieve a material balance. For the whole block method, the diluted model includes revised diluted density and metal grades for all blocks.

AGP also believed that contact dilution would play a role in material sent to the mill. The size of the block in the model was examined as well as the ore zone thicknesses to determine the amount of dilution that would be appropriate for the SW deposit. A dilution skin thickness of 0.4 m was used for SW and resulted in the diluted mill feed containing 9.0% more tonnes and 6.8% lower gold grades than the in-situ mill feed summary. AGP considered this dilution as acceptable due to the narrow mineralization zones being mined.

The resource model provided for the north deposit (87, J and X22 zones) was also a whole block, grade model which includes some internal dilution. Similar to the SW model, AGP believed that this did not adequately reflect the amount of dilution that would be expected with normal mining practice, even with more selective equipment A dilution skin thickness of 0.4 m was used for the north deposits and resulted in the diluted mill feed containing 6.2% more tonnes and 4.6% lower gold grades than the in-situ mill feed summary. AGP considered this dilution as acceptable due to the wider mineralization zones being mined.

A mining recovery of 98% was applied in the final diluted mill feed summaries to account for losses in mining and transportation to the mill.

15.6 Pit Design

Pit designs were developed for the J, 87, X22 and SW pit areas. The J pit design consists of two phases of successive pushbacks around the entire pit perimeter. The 87-pit design includes an initial phase 0 at the south end to assist with site water management, followed by phases 1 to 3 in the main portion of the pit. The X22 pit design consists of two phases, with slightly higher grades in the phase 1 at the south. The SW pit design consists of two phases which can be scheduled as satellite phases from the northern pits. The pit optimization shells used to guide the ultimate pits were also used to outline areas of higher value for targeted early mining and phase development. All pits were developed using 10 metre bench heights.

Tonnes and grade for the designed pit phases are reported in Table 15-4 using the diluted tonnes and grade from the models and a mining recovery of 98% to account for additional mill feed losses.

Table 15-4: Pit Phase Tonnages and Grades

Pit	Phase	Ore (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	NSR (C$/t)	Waste (Mt)	Total (Mt)	Strip Ratio
J	1	74.4	0.45	0.06	0.91	29.53	153.0	227.4	2.1
	2	50.8	0.42	0.058	0.84	27.79	164.7	215.5	3.2
J Total		**125.2**	**0.44**	**0.058**	**0.88**	**28.82**	**317.7**	**442.9**	**2.5**
87	0	1.6	0.65	0.04	0.95	42.20	8.5	10.1	5.3
	1	31.6	0.55	0.062	1.17	37.09	139.3	170.9	4.4
	2	69.0	0.58	0.068	1.14	39.38	179.5	248.5	2.6
	3	63.9	0.52	0.055	1.08	34.26	272.0	335.9	4.3
87 Total		**166.1**	**0.55**	**0.062**	**1.12**	**37.00**	**599.4**	**765.5**	**3.6**
X22	1	16.5	0.43	0.07	1.61	29.59	56.5	73.0	3.4
	2	20.0	0.40	0.047	0.79	25.48	53.1	73.0	2.7
X22 Total		**36.4**	**0.41**	**0.058**	**1.16**	**27.34**	**109.6**	**146.0**	**3.0**
SW	1	34.0	0.48	0.05	0.75	29.09	75.1	109.0	2.2
	2	17.9	0.52	0.035	0.78	30.67	69.2	87.1	3.9
SW Total		**51.9**	**0.49**	**0.045**	**0.76**	**29.64**	**144.3**	**196.1**	**2.8**
Troilus Total		**380**	**0.49**	**0.058**	**1.00**	**32.37**	**1,171**	**1,550**	**3.1**

15.7 Mine Schedule

The mine schedule for open pit mining consists of 380 Mt of mill feed grading 0.49 g/t gold, 0.058% copper, and 1.0 g/t silver providing mill feed for 22 production years. Open pit waste tonnage totals 1,171 Mt and will be placed into waste storage areas. The overall open pit strip ratio is 3.1:1. The mine schedule utilizes the pit phases described previously to send a maximum of 18.3 Mtpa (50,000 tonnes per day) of feed to the mill facility.

The current mine life includes two years of pre-stripping followed by twenty-one years of mining. A maximum descent rate of 9 benches per year per phase was applied for open pit mining to ensure that reasonable mining operations and mill feed control would occur. Peak primary mining rates are scheduled at approximately 87 Mt per year between years 5 to 8. Mill feed is stockpiled during the pre-production years, with approximately 0.7 Mt of feed for plant commissioning. A peak stockpile capacity of 48 Mt was reached near the end of year 11. Stockpile material is reclaimed from stockpiles after completion of mining and continues until early into the 22nd year.

15.8 Mineral Reserves Statement

The reserves for the Project are based on the conversion of the Measured and Indicated Mineral Resources in the current mine plan within the J, 87, X22, and SW open pits. No Measured Mineral Resources are contained in these deposits, so there will be no Proven Reserves. Indicated Mineral Resources are converted directly to Probable Reserves.

The Mineral Resources in the underground areas below the pits are also not considered in the Mineral Reserves at this time. The total Mineral Reserves for the Project are shown in Table 15-5. Some variation may exist due to rounding.

Table 15-5: Proven and Probable Reserves – January 15, 2024

Reserve Class	Tonnage (Mt)	Grades Au (g/t)	Cu (%)	Ag (g/t)	AuEq (g/t)	CuEq (%)	Contained Metal Au (Moz)	Cu (Mlb)	Ag (Moz)	AuEq (Moz)	CuEq (Blb)
Proven	0	0.00	0.000	0.00	0.00	0.00	0.00	0	0.00	0.00	0.00
Probable	380	0.49	0.058	1.00	0.59	0.39	6.02	484	12.15	7.26	3.24
Proven and Probable	**380**	**0.49**	**0.058**	**1.00**	**0.59**	**0.39**	**6.02**	**484**	**12.15**	**7.26**	**3.24**

Note: This mineral reserve estimate has an effective date of January 15, 2024, and is based on the mineral resource estimate dated October 2, 2023, for Troilus Gold by AGP Mining Consultants Inc. The mineral reserve estimate was completed under the supervision of Willie Hamilton, P.Eng. of AGP, who is a Qualified Person as defined under NI 43-101. Mineral Reserves are stated within the final pit designs based on a US$1,550/oz gold price, US$20.00/oz silver price and US$3.50/lb copper price. An NSR cut-off of C$9.96/t was used to define reserves. The life-of-mine mining cost averaged C$3.99/t mined, preliminary processing costs were C$8.02/t ore and G&A was C$1.94/t ore placed. The metallurgical recoveries were varied according to gold head grade and concentrate grades. 87 pit recoveries for equivalent grades were 95.5%, 94.7% and 98.2% for gold, copper, and silver respectively. J pit recoveries for equivalent grades were 93.1%, 89.3% and 88.9% for gold, copper, and silver respectively. X22 pit recoveries for equivalent grades were 95.5%, 94.7% and 98.2% for gold, copper, and silver respectively. SW pit recoveries for equivalent grades were 85.7%, 91.5% and 85.6% for gold, copper, and silver respectively. The formulas used to calculate equivalent values are as follows, for 87 Pit AuEq = Au + 1.5361*Cu +0.0133 *Ag, for J Pit AuEq = Au + 1.4849*Cu +0.0123 *Ag, for SW Pit AuEq = Au + 1.6535*Cu +0.0129 *Ag, for X22 Pit AuEq = Au + 1.5361*Cu +0.0133 *Ag.

15.9 Factors That May Affect Mineral Reserve Estimate

The QP has not identified any known legal, political, environmental, or other risks that would materially affect the mineral reserve estimates.

Risks that could materially affect the reserve include mining selectivity near the ore contacts, slope stability and assumed process recoveries for given rock types. These are considered manageable risks which will be mitigated as more testwork, and operating experience is obtained.

16 MINING METHODS

16.1 Introduction

The Project is located in central Québec and is situated approximately 120 km north of Chibougamau. The Z (or Z87) Zone and J (or J4/J5) Zone were subject to open pit mining operations between 1996 to 2010. It has been established that there are still significant open pit and underground mineral resources in these zones.

The Mineral Resources for the Project include the four principal mineralized zones: 87, J, X22 and SW Zones. AGP's opinion is that with current metal pricing levels and knowledge of the mineralization and previous mining activities, open pit mining offers the most reasonable approach for development of all zones. This is based on the size of the resource, tenor of the grade, grade distribution and proximity to topography for the deposits. Underground mining remains as a possible supplement to mill feed tonnes and grade.

The mine schedule for open pit mining consists of 380 Mt of mill feed grading 0.49 g/t gold, 0.058% copper, and 1.0 g/t silver over a processing life of slightly more than 21 years. Open pit waste tonnage totals 1,171 Mt and will be placed into waste storage areas. The overall open pit strip ratio is 3.1:1. The mine schedule utilizes open pit mining areas to supply mill feed up to a maximum of 18.3 Mtpa to the mill facility.

The current mine life includes two years of pre-stripping followed by twenty-one years of production mining. Mill feed is stockpiled during the pre-production years and reaches a peak stockpile capacity of 48 Mt near the end of year 11.

The open pit mining starts in Year -2 and continues uninterrupted until Year 21.

16.2 Mining Geotechnical

Troilus retained WSP to provide Feasibility level (FS) open pit slope design recommendations for the four principal mineralized zones of the Project, 87, J, X22 and SW. The pit slope geotechnical assessments are detailed in the design reports 22538260-007-R-Rev0 dated 21 June 2024 (87, J and SW pits) (WSP, 2024a) and 22538260-010-R-Rev0-6000 dated 20 June 2024 (X22 pit) (WSP, 2024b).

The pit slope recommendations are based on observations from the historical slope performance; previous consultant reports; the 2020, 2021 and 2023 geotechnical investigations completed by WSP, and the 3D interpretation of geological units and major structures prepared by Troilus. Oriented data from exploration holes was also used to assess foliation spatial variations and confirmed the design assumptions derived from other data sources.

The Inmet experience in mining 87 and J pits between 1996 and 2009 provides an excellent source of information to develop evidence-based pit slope designs. This experience improves our confidence in the design as it provides data on slope behaviour including continuity of structures and backbreak related to the interaction between operational practices and bench scale rock fabric. The bench face angles achieved on the various walls by Inmet were reviewed and used as a benchmark to assess the crest backbreak and the structural controls for walls with similar orientations and rock quality.

Topographical surveys of the J Pit (March 2008) and 87 Pit (April 2009) with a 2 m resolution were provided by Troilus in Fall 2021. This information was used to document and understand achieved bench face angles at 817 locations in the 87 Pit and 279 locations in the J Pit. WSP also conducted two geotechnical investigations to characterize the rock mass quality and structural fabric of the planned pit walls. The first geotechnical campaign, completed in 2020- 2021, included wall mapping and photogrammetry of the 87 Pit, logging of sixteen (16) geotechnical boreholes and thirty-two (32) hydraulic conductivity tests in eleven (11) boreholes and geophysical logging of seven (7) boreholes. The second geotechnical campaign was completed in 2023 to fill data gaps and included logging of three (3) geotechnical boreholes with thirteen (13) hydraulic conductivity tests. As of 2022, Troilus began oriented core drilling of their exploration holes. A total of 92 oriented boreholes with 3539 oriented structures across all four deposits were made available to WSP by Troilus. Data sources on historical performances include observations from the historical operations, previous consultant reports and studies, Inmet monitoring, ground support and fall of ground data.

The geotechnical investigations indicate that the ultimate pit walls will expose mainly Very Good Quality rock mass (RMR>80) with high intact rock strength (>100 MPa), which is the same as the rocks previously exposed at 87 and J. The oriented data and recent Troilus structural model indicate structural fabric in the planned walls similar to what was previously observed.

Slopes were assessed for kinematic planar, wedge, and toppling failures by wall orientation. Historical prism measurements were reviewed to assess displacement, helping to calibrate numerical modelling for the west walls. Finite element (RS2), discrete element (UDEC) and limit equilibrium toppling (Roctopple) analyses were run to determine rock slope designs for the west walls. Limit equilibrium assessments, including bi linear slab analyses to check for failure along anisotropies were completed for the east walls. The fall of ground events and ground support installed on the pit walls were reviewed to provide insight on the operational risks that are expected to repeat on similar new walls.

Records of ground support installed by Inmet at the J and 87 open pits to develop steeper double benches and steeper inter-ramp angles were compiled and reviewed. This information was used to estimate quantities for ground support in analogous slope design domains in the proposed new open pits, considering historical bolting requirements and the results of kinematic assessments.

The historical instabilities that occurred between 2000 and 2009 were well-documented by Inmet. Around 65 failures occurred in the 87 pit for a total of 97 084 tons and 5 failures in the J pit, for a total of 4 920 tons of material. Some of these failures interrupted operations.

The stability analyses completed for this study supported by the historical operation experience in mining pit 87 and J indicate that the rock mass is favourable to the development of steep inter-ramp slopes in all areas of the pits except the east walls where the bench face angle is constrained by planar failure involving moderately dipping foliation. Specific conclusions supported by the characterization and analyses described by the geotechnical assessments include:

- **West Wall:** At 87 pit: Stability analyses calibrated on historical displacements and failures observed on the west walls of the J and 87 pits indicate increasing risks of toppling and planar failure with pit deepening. The pit deepening will induce slope relaxation that results in upper slope kinematic failures (planar, wedge, with toppling release) developing on the west walls. The recommended inter-ramp, overall slope angles and accompanying maximum slope heights of these walls will need to be respected to ensure wall stability and that the mining can be completed without developing deep- seated toppling or significant operational delays. On previous slopes, pit deepening led to increased rockfalls and ravelling resulting in restricted or lost access and suspension of mining.

 - Overall slope stability will also be very sensitive to water pressure past certain depths. Pro-active depressurization with horizontal drains will be required to implement the slope design. If monitoring indicates less than adequate slope depressurization or if it shows signs of deep-seated deformation occurring at overall slope heights shallower than predicted, step-ins, pushbacks or other remedial measures will be required sooner than proposed.

- **East Walls:** A review of the existing 87 and J bench geometries indicates that the benches backbreak to the mean foliation dip. For the design, the 50% cumulative frequency of foliation dip measured in core and televiewer was used to define the bench face angle. A 50% cumulative frequency is on the aggressive side of the typical range used in the industry. Limit equilibrium assessments, including bi linear slab analyses to check for failure along anisotropies were completed for the east walls.
  - Similar to the Inmet experience, ground support will be required where the design bench face angle is undercut by continuous open features or tapered wedges. Careful perimeter blasting and scaling will also be required to successfully apply this design and reduce backbreak associated with blasting.
- **In other pit sectors**, where high risks of toppling failures and planar failures on foliation are not a concern, steep bench face angles should generally be achievable with careful blasting, excavation, and scaling. While wedge or toppling failures may occur locally, available structural data does not indicate these failure mechanisms to be a widespread control on bench design. The design bench face angles are defined based on the achieved bench face angles at 87 and J in similar rock mass domains and validated by the kinematic assessment of structures.
- **Locally, zones** of intense fracturing that may be encountered in association with faulting, dykes or contacts and that could result in a decrease in stable bench face angles are currently understood to be relatively narrow and will affect bench stability over only limited lengths that should not require major slope redesign. Additional ground support may be required in those zones.
- **While the past** operational experiences suggest that pit dewatering should result in generally depressurized slopes, this should be verified by monitoring, particularly for the hanging wall slope where bench stability may be sensitive to groundwater pressures and anisotropy may affect drainage characteristics. In other sectors, groundwater is not expected to be a control of rock mass stability, and slope dewatering required for pit operations is likely to be sufficient for maintaining rock mass stability.

16.2.1 Recommended Slope Configurations

Our pit slope designs were optimized assuming that ground support will be used to achieve safe bench geometries, building on the precedents and experience gained by Inmet operation so doing while mining the historical 87 and J pits. The implementation of the slope designs in Table 16-1 will require a dedicated team, strong procedures and systems and capability to adapt. It will require a high level of skill applied to perimeter blasting, scaling, ground support, depressurization and slope movement monitoring by the mine team.

Table 16-1: Recommended Open pit Slope Angles for the Troilus Project [1, 2, 3, 4, 6]

Pit	Domain	Wall	Effective Bench Face Angle (°)	Effective Catch Bench Width (m)	Design Pre-split Angle (°)	Design Catch Bench Width (m)	Inter-Ramp Slope Angle (°)	Maximum Overall Slope Angle (°)	Depressurization Approach[4]	Ground Support Requirements[5]
All	NA	Overburden	2.5(H):1(V)	7	NA	NA	3(H):1(V)	NA	NA	NA
J	III-A	North of East (210° - 320°)	45	8.5	60	16.9	35.1[7]	NA	Passive Depressurization	1 bolt/200 m^2
	III-A	Upside North of East (210° - 320°)	56	8.5	68	14.0	42.2[7]	NA	Passive Depressurization	1 bolt/50 m^2
	III & III-B	East (210° - 320°)	63	8.5	68	10.6	47.0[7]	NA	Passive Depressurization	1 bolt/200 m^2
	I-A	North of West (90° - 135°)	68	7.5	90	15.6	Up to 100 m = 56.6 ° Up to 150 m = 52 °	43[9]	Passive Depressurization	1 bolt/200 m^2
	I	South of West (90° - 135°)	74	7.5	90	13.2	Up to 150 m = 56.6 °	43[9]	Passive Depressurization	1 bolt/200 m^2
	I	Upside of South of West (90° - 135°)	74	7.5	90	13.2	Up to 150 m = 56.6 °	45[9]	Enhanced Depressurization	1 bolt/200 m^2
	II	North (135° - 210°)	74	7.5	90	13.2	56.6	NA	Passive Depressurization	1 bolt/200 m^2
	IV	South (320° - 90°)	72	7.5	90	14.0	55.0	NA	Passive Depressurization	1 bolt/200 m^2
87	I	West (80° - 140°)	68	7.5	90	15.6	Up to 150 m = 56.6 ° Up to 200 m = 52 °	41[8]	Passive Depressurization	1 bolt/1000 m^2
	I	Upside for West	74	7.5	90	13.2	Up to 200 m = 56.6 °	43[8]	Enhanced Depressurization	1 bolt/200 m^2
	III	North-west (140° - 205°)	75	7.5	90	12.9	57.2	NA	Passive Depressurization	1 bolt/1500 m^2
	IV	North-east (205° - 265°)	66	7.5	90	16.4	50.6	NA	Passive Depressurization	1 bolt/300 m^2
	V	East (265° - 000°)	60	8.5	65	10.7	45.0[7]	NA	Passive Depressurization	1 bolt/200 m^2
	VI	South (000° - 80°)	71	7.5	90	14.4	54.2	NA	Passive Depressurization	1 bolt/100 m^2

Table 16-1 (continued): Recommended Open pit Slope Angles for the Troilus Project[1, 2, 3, 4, 6]

Pit	Domain	Wall	Effective Bench Face Angle (°)	Effective Catch Bench Width (m)	Design Pre-split Angle (°)	Design Catch Bench Width (m)	Inter-Ramp Slope Angle (°)	Maximum Overall slope Angle (°)	Depressurization Approach[4]	Ground support requirements
SW [10]	I	West (090° - 200°)	70	7.5	90	14.8	Up to 150 m = 53.5	45[9]	Passive Depressurization	1 bolt/1000 m^2
	I	Upside for West (090° - 200°)	74[8]	7.5	90	13.2	Up to 200 m = 56.6	48[9]	Enhanced Depressurization	1 bolt/200 m^2
	II	North (130° - 265°)	68	7.5	90	15.6	52.0	NA	Passive Depressurization	1 bolt/200 m^2
	III	East (265° - 000°)	68	8.5	73	10.5	50.3 [7]	NA	Passive Depressurization	1 bolt/100 m^2
	IV	South (000° - 090)	70	7.5	90	14.8	53.5	NA	Passive Depressurization	1 bolt/200 m^2
X22		South-East	50	8.5	50[11]	8.5	38.3 [7]	NA	Passive Depressurization	1 bolt/200 m^2
		North-West	72	7.5	90	14	55	NA	Passive Depressurization	1 bolt/200 m^2
		South-West	70	7.5	90	14.8	53.5	NA		1 bolt/200 m^2

Notes

1) Bench Configuration: Double bench - 2 x 10 m
2) Incorporate a haul road or additional 15 m width for a geotechnical bench below slopes 200-250 m high to de-couple the slopes or if the west wall design does not meet the maximum interim height or overall slope angle criteria.
3) Avoid convex shapes and integrate design transition in the steepest domain.
4) The target for enhanced depressurization is a water table located 40 m behind the wall and the assumption for passive depressurization (base case) is 15 m behind the wall.
5) For planning purposes assume 6 m long 50-ton rock bolts. Optimization of the inclination of the pre-split may be required to limit ground support requirements.
6) For phase walls, add 3 m to the design catch bench width.
7) Local adjustments to slope design may be required where micro folding not captured in the current interpretation occur. Where foliation is observed to provide the main structural control, the effective BFA should be adjusted to follow the dip angle of foliation.
8) For a maximum slope height of 450 m.
9) For a maximum slope height of 300 m.
10) The effective BFA considered for the upside design of the west wall is considered very aggressive based on the current amount of information available at SW pit. This design should not be attempted without collecting additional geotechnical information to confirm rock quality in this wall.
11) Will be excavated with controlled blasting to break cleanly along structures.

16.2.2 Future Geotechnical Work

In discussion with Troilus, it was decided to exclude SW pit from the 2023 field investigation because it was thought at the time that this pit would be mined last. SW pit is now planned to be mined in the first years of the operation. Discussions on data uncertainty at SW pit are included in the pit slope design report for this pit. Additional geotechnical drilling is recommended in this pit with a focus on characterization of structures. An assumed conservative design was recommended for this pit in the absence of data.

Continue collecting orientated structural data in exploration holes. Foliation data near east walls will be key in understanding local folding to plan for local adjustments to the pit design. Investigate the sharp variation in foliation orientation observed between exploration holes located close to each other when limited changes are observed within single holes.

During detail design, complete a review of the mining sequence to validate that the overspill hazard is managed, and that mining will never occur simultaneously on two levels of the same wall. Also, no critical infrastructure should be located within 60 m of the 87-pit west wall crest to allow for an offload of the crest if the lower bench is considered unstable with the current design.

Plan for equipment, manpower, delays and costs associated with excellent wall protection and wall monitoring practices as well as extensive ground support installation described in the previous section. Plan for radar monitoring of the west walls where mining sequencing between phases will develop highest walls.

16.3 Open Pit

16.3.1 Geologic Model Importation

The 2023 resource estimates for the 87 and SW deposits were created using Geovia Surpac software for mineralization domains and block modelling, while Leapfrog Geo and Edge were used for the J and X22 deposits. The 87, J and X22 models were combined into a single block model for the north area of the project. CSV block model format files were created as whole block models for use in open pit mine planning tasks.

Framework details of the two open pit block models are provided in Table 16-2. Significant mine planning model items for SW and North models are displayed in Table 16-3 and Table 16-4 respectively. The mining models created by AGP in Hexagon MinePlan® includes additional items for mine planning purposes. MinePlan® was used for the mining portion of the FS, utilizing their Lerchs Grossman (LG) shell generation, pit and dump design and mine scheduling tools. Only Indicated resources were used for conversion to reserves in the FS. No Measured resources were reported in the models provided.

Table 16-2: Open Pit Model Frameworks

Framework Description	SW model Value	North model Value
MinePlan® file 10 (control file)	sw2310.dat	np2310.dat
MinePlan® file 15 (model file)	sw2315.m01	np2315.m01
X origin (m)	8,400	8,730
Y origin (m)	7,650	11,600
Z origin (m) (max)	5500	5500
Rotation (degrees clockwise)	0	0
Number of blocks in X direction	440	454
Number of blocks in Y direction	870	920
Number of blocks in Z direction	220	200
X block size (m)	5	5
Y block size (m)	5	5
Z block size (m)	5	5

Table 16-3: SW Model Item Descriptions

Field Name	Min	Max	Precision	Units	Comments
ROCK	0	999	1	-	Rock types
DEN	0	9	0.01	t/m^3	Bulk Density
AU	0	99	0.001	g/t	Gold grade
AG	0	99	0.001	g/t	Silver grade
CU	0	9	0.0001	%	Copper grade
CLASS	0	9	1	-	Resource Class (1=mea, 2=ind, 3=inf, 4=unclassified, 9=default)
LITH	0	99	1	-	Lithology
TOPO	0	100	0.01	%	Percentage below topography
RTYPE	1	10	1	-	Rocktype code (1= overburden, 2=backfill, 3=bedrock)
RSCOD	-9	9	1	-	3D mining restriction, used for TSF 30m offset
PIT	0	9	1	-	Pit number, 3 = SW pit (for NSR calculations)
MCO35	0	9	0.01	C$/t	Ore mining cost for pit optimization
MCW35	0	9	0.01	C$/t	Waste mining cost for pit optimization
NSR3	0	9999	0.01	C$/t	NSR for MI pitshell with in-situ grades
DAU	0	99	0.001	g/t	diluted gold grade
DCU	0	9	0.0001	%	diluted copper grade
DAG	0	99	0.001	g/t	diluted silver grade
BLOKT	0	999	0.01	t	Insitu block tonnes
DTON	0	999	0.01	t	Diluted block tonnes
DDEN	0	99	0.001	g/cc	Diluted Bulk Density
OWFL	0	1	1	-	Ore waste flag for dilution, where 0=waste and 1=ore
NSR3D	0	9999	0.01	C$/t	NSR for MI pitshell with diluted grades
VLT3	0	999	0.01	C$/t	Value per tonne using NSR3D
VLB3	9999	1000000	1	C$	Net block value using NSR3D
DISC	0	1	0.001	-	Discount factor by depth
SWMET	0	30000	1	-	Met sample values, 800 kg = default, otherwise 3000 kg

Table 16-4: North Model Item Descriptions (87, J and X22)

Field Name	Min	Max	Precision	Units	Comments
DEN	0	9	0.01	t/m^3	Bulk Density
AG	0	99	0.0001	g/t	Silver grade
AU	0	99	0.0001	g/t	Gold grade
CU	0	9	0.0001	%	Copper grade
LITH	0	999	1	-	Lithology
ROCK	0	9999	1	-	Rock Type (1=OB,2=Backfill,3=ROCK)
CLASS	0	9	1	-	Resource Class (1=mea, 2=ind, 3=inf, 4=unclassified, 9=default)
PIT	0	9	1	-	Pit number, (for NSR calculations), 1=J, 2=87, 4=X22), 0=air
TOPOR	0	100	0.01	%	Percentage below topography (before backfilling in J)
TOPO	0	100	0.01	%	Percentage below topography
RTYPE	0	9	1	-	Rock Type (1=OB,2=Backfill,3=ROCK)
BFPCT	0	100	0.01	%	Percentage of block within J backfill
WDPCT	0	100	0.01	%	Percentage of block within waste dumps
MCO35	0	9.99	0.01	C$/t	Ore mining cost for pit optimization
MCW35	0	9.99	0.01	C$/t	Waste mining cost for pit optimization
SLOPE	0	99	1	-	Slope domains, where 1= OB, 2=BF or WD, 31=J rock, 32=87 rock, 34=X22 rock
BLOKT	0	999	0.01	t	Insitu block tonnage
OWFL	0	1	1	-	Ore waste flag for dilution, where 0=waste and 1=ore
DTON	0	999	0.01	t	Diluted block tonnage
DDEN	0	99	0.01	t/m^3	Diluted density
DAU	0	99	0.0001	g/t	Diluted Au grade
DAG	0	99	0.0001	g/t	Diluted Ag grade
DCU	0	9	0.0001	%	Diluted Cu grade
PITM	0	9	1	-	Adjusted Pit number, (1=J, 2=87, 3=SW, 4=X22), topo blocks corrected
NSR4	0	9999	0.01	C$/t	NSR for MI pitshell with in-situ grades
NSR4D	0	9999	0.01	C$/t	NSR for MI pitshell with diluted grades
VLT4D	0	999	0.01	C$/t	Value per tonne using NSR4D
VLB4D	-9999	1000000	1	C$	Net block value using NSR4D
DISC	0	1	0.001	-	Discount factor by depth
SWMET	0	30000	1	-	Met sample values, 800kg = default, otherwise 3000 kg

16.3.2 Economic Pit Shell Development

The open pit ultimate size and phasing opportunities were completed with various input parameters including estimates of the expected mining, processing, and G&A costs, as well as metallurgical recoveries, pit slopes, and reasonable long-term metal price assumptions. AGP worked together with Troilus personnel to select appropriate operating cost parameters for the Troilus open pits.

Wall slopes for pit optimization were based on the assessment discussed in Section 16.2. Slopes were flattened as required due to inclusion of haulage ramps. The overall slope angles for use in LG routines are shown in Table 16-5.

Table 16-5: Pit Shell Slopes

Pit	Slope	Start Azim. (degrees)	Material Type	Overall Slope (degrees)	Height (m)	Inter-ramp angle (degrees)	36.5 m roads	Extra berms
J	All slopes		OB, fill	17.5	90	21.8		1
	East	357	Bedrock	43	395	47	1	1
	South	107	Bedrock	47	360	55	2	
	West	237	Bedrock	46	415	52	2	
	North	282	Bedrock	49	360	57	2	
87	All slopes		OB, fill	17.5	90	21.8		1
	East	52	Bedrock	39	475	45	3	
	South	147	Bedrock	48	465	54	2	
	West	227	Bedrock	44	460	52	3	
	Northwest	287	Bedrock	48	470	57	3	
	Northeast	352	Bedrock	43	475	51	3	
X22	All slopes		OB, fill	17.5	90	21.8		1
	Northwest	267	Bedrock	46	200	55	1	1
	Southeast	47	Bedrock	40	200	47	1	1
	South & West	177	Bedrock	46	200	55	1	1
SW	All slopes		OB, fill	21.8		21.8		
	East	52	Bedrock	41	280	50	2	1
	South	147	Bedrock	45	280	54	2	
	West	237	Bedrock	47	280	56.5	2	
	North	347	Bedrock	47	280	52	1	

The mining costs are estimates based on cost estimates for equipment from vendors and previous studies completed by AGP. The costs represent what is expected as a blended cost over the life of the mine for all material types to the various dump locations. Process costs and a portion of the G&A costs were developed by AGP in consultation with Troilus and Lycopodium personnel.

The parameters used are shown in Table 16-6. Costs and revenues are in United States dollars for use in pit shell determination unless otherwise noted. The mining cost estimates are based on the use of 227 tonne trucks using an approximate waste dump configuration to determine incremental hauls for mill feed and waste. Copper, gold, and silver are the primary elements used in the revenue calculations.

The smelting terms and recovery assumptions are based on creating a specific copper bulk concentrate for each pit. The target copper concentrate grades for J, 87, X22 and SW pits in this study were 15%, 18%, 15% and 16% respectively when the copper feed grade was less than or equal to 0.15%. When the copper feed grade was above 0.15%, the target copper concentrate grades for J, 87, X22 and SW pits in this study were 18%, 22%, 18% and 20% respectively.

Table 16-6: Economic Pit Shell Parameters (US Dollars unless otherwise noted)

Description Exchange rates CAD	Units US$ =	Value 1.30		
Metal Prices		**Copper**	**Gold**	**Silver**
Price	$/oz	3.50	1550.00	20.00
Royalty	%	1.0%	1.0%	1.0%
Dore				
Payable	%		99%	99.0%
Selling Cost	$/oz		12.00	1.00
Smelting, Refining, Transportation Terms				
Payable	%	96.6%	98.0%	97.0%
Minimum Deduction	unit, g/dmt	1.1	2	20
Participation (on profits)	%	100%	100%	100%
Smelting Charge	$/dmt	99.00		
Refining	$/oz, $/lb	0.099	5.00	0.50
Concentrate Moisture	%	8%		
Transit Losses	%	0.5%	0	0
Concentrate Trucking Cost	$/wmt	92.27		
Concentrate Port Cost	$/wmt	20.00		
Concentrate Shipping Cost	$/wmt	65.00		
Metallurgical Information				
		Copper	**Gold**	**Silver**
Tails solid assay (TG)		%	g/t	g/t
J zone		0.0050	0.0320	0.1030
SW zone		0.0045	0.0630	0.1285
87 zone		0.0050	0.0290	0.0275
X22 zone		0.0050	0.0320	0.1030
Gravity recovery (GR)		Copper	Gold	Silver
J zone	%	0	33.18	33.18
SW zone	%	0	22.48	22.48
87 zone	%	0	36.33	36.33
X22 zone	%	0	30.66	30.66
Copper Concentrate Grade		**Cu Feed Grade**	**Cu Concentrate Grade**	
		Cutover value	<= cutover	> cutover

Description Exchange rates CAD	Units US$ =	Value 1.30		
J zone	%	0.15	15	18
SW zone	%	0.15	16	20
87 zone	%	0.15	18	22
X22 zone	%	0.15	15	18
Losses for Gravity circuit	%	0.025		
Power Cost				
Cost of power	C$/Kwhr	0.035		
Fuel Cost				
Diesel Fuel Cost to site	C$/ l	1.66		
Mining Cost *				
Base Rate – 5,360 m Elevation		**J and X22 Zones**	**87 Zone**	**SW**
Waste	C$/t moved	2.15	1.99	2.01
Mill Feed	C$/t moved	2.29	2.10	2.37
Incremental Rate - below 5360 m				
Waste	C$/t moved/ 10 m bench	0.039	0.041	0.036
Mill Feed	C$/t moved/ 10 m bench	0.036	0.029	0.028
Processing and G&A				
Processing Cost	C$/t mill feed	8.02		
G&A Cost	C$/t mill feed	1.94		
Process + G&A	C$/t mill feed	9.96		

* mining costs based on using 227 t haul trucks

Nested LG pit shells were generated to examine sensitivity to metal prices with base case prices of US$1,550/oz Au, US$3.50/lb Cu and US$20.00/oz Ag. This was done to gain an understanding of the deposit and highlight potential opportunities in the design process to follow. Diluted Indicated resource material was used in the final analysis. The net smelter return (NSR) was varied by applying revenue factors of 0.10 to 1.20 at 0.05 increments, to generate a set of nested LG shells. The chosen set of revenue factors result in an equivalent gold price varying from US$150/oz up to US$1,800/oz. All other parameters were fixed. The resulting nested pit shells assist in visualizing natural breakpoints in the deposit and selecting shells to act as design guidance for phase design. The net profit before capital for each pit was calculated on an undiscounted basis for each pit shell using the above base case prices. Mill feed/waste tonnages and net profit were plotted against equivalent gold price and are displayed in the next few figures.

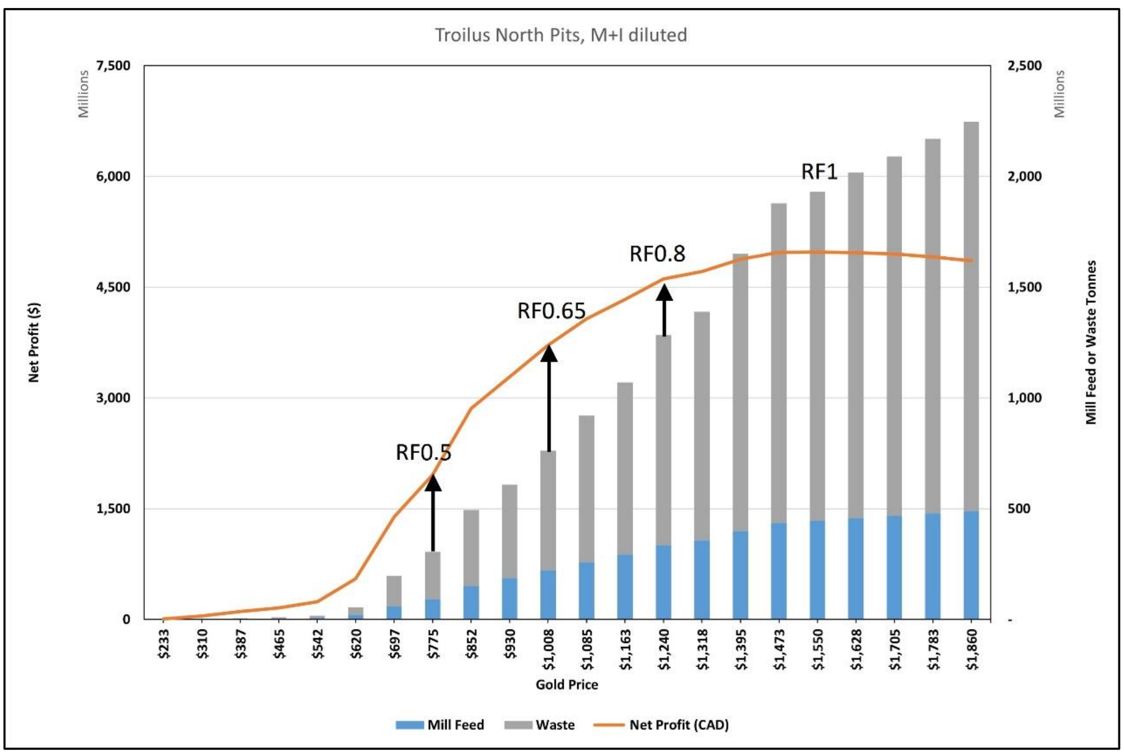
The plot of the north pits (J, 87 and X22) profit versus price is displayed in Figure 16-1. The three pits were run together in order to capture the benefit of shared stripping in the adjacent pits. With each incremental increase in the waste tonnage, and to a lesser degree the mill tonnage, the undiscounted net profit also increased. In the case of the first break point shown at US$775/oz Au, the cumulative waste tonnage is 214 Mt, with a corresponding mill feed tonnage of 91 Mt or a strip ratio of 2.3:1. The net profit also increased beyond this point showing that there was still value to be obtained by going with a higher metal price or an additional phase. This break point represented 39% of the net value of a $1,500/oz pit but with only 14% of the waste of the larger pit shell.

The second break point was at US$1008/oz Au. The incremental waste tonnage from the first break point is 325 Mt, with a corresponding increase in mill feed tonnage of 132 Mt or a strip ratio of 2.5:1. The cumulative value of the first two break points was 75% of the US$1,500/oz Au pit shell but with only 36% of the waste movement of the larger pit required.

The third and final break point was at US$1240/oz Au. The incremental waste tonnage from the second break point is 623 Mt, with a corresponding increase in mill feed tonnage of 203 Mt or a strip ratio of 3.1:1. The cumulative value of the first three break points was 93% of the US$1,500/oz Au pit shell but with only 64% of the waste movement of the larger pit required.

The last pit shell ran the entire length of the orebodies and allows a reasonable mining width to early phases where appropriate. The additional potential pit value in larger pit shells was considered insufficient to cover schedule discounting. Particular attention would need to be taken to ensure haul road access was available to each of the phases at various elevations.

Figure 16-1: North Pits Profit vs. Price by Pit Shell

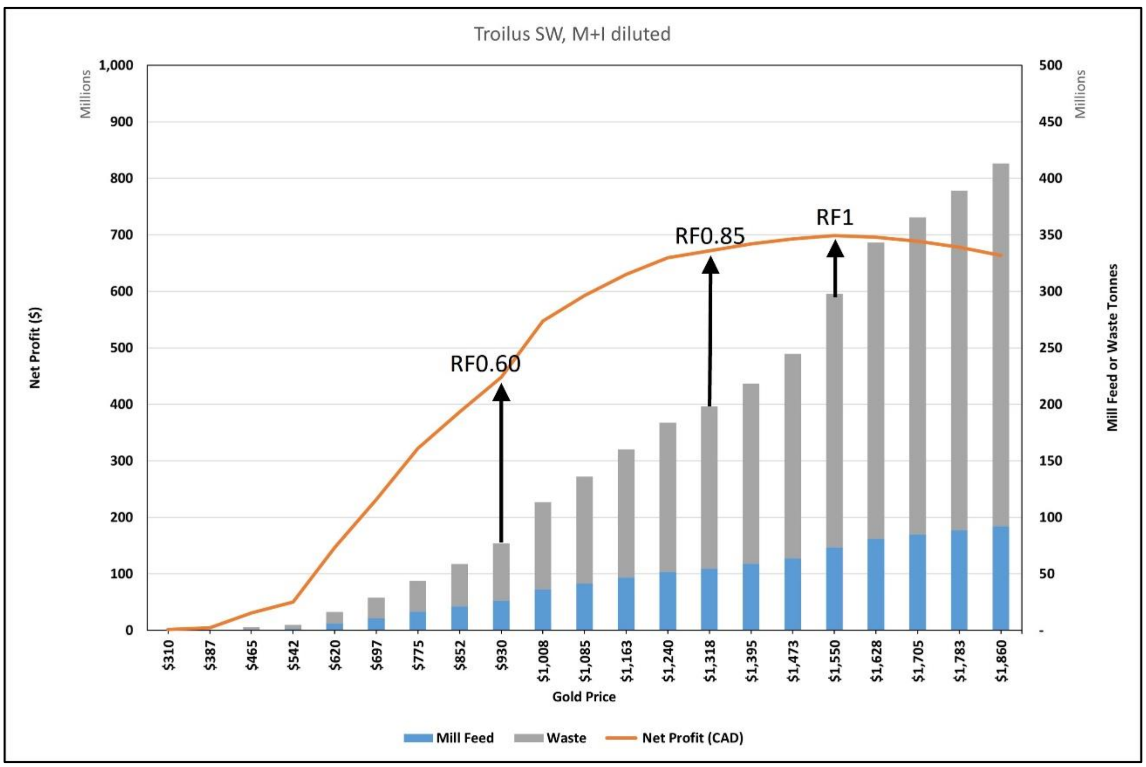


Source: AGP (2024)

The plot of SW pit profit versus price is displayed in Figure 16-2 and illustrates various break points in the pit shells. A restriction was placed on the pit optimization run so that pit shells could not expand within a 30-metre offset of the anticipated tailings storage facility toe. In the case of the first break point shown at US$930/oz Au, the cumulative waste tonnage is 50.6 Mt, with a corresponding mill feed tonnage of 26.4 Mt or a strip ratio of 1.9:1. The net profit also increased beyond this point showing that there was still value to be obtained by going with a higher metal price or an additional phase. This break point represented 64% of the net value of a $1,500/oz pit but with only 23% of the waste of the larger pit shell.

The next significant break point was at US$1318/oz Au. The incremental waste tonnage from the first break point is 93.0 Mt, with a corresponding increase in mill feed tonnage of 28.3 Mt or a strip ratio of 3.3:1. This pit shell was used for the design of the phase 2, or ultimate pit. The cumulative value of the final selected pit shell was 96% of the US$1,500/oz Au pit shell but with only 64% of the waste movement of the larger pit required. This pit shell ran the entire length of the orebody and allows a reasonable mining width from phase 1. The additional potential pit value in larger pit shells was considered insufficient to cover schedule discounting.

Figure 16-2: SW Pit Profit vs. Price by Pit Shell



Source: AGP (2024)

16.3.3 Dilution

Both the north pit and SW resource models were provided in a whole block format. Whole block models means that for any given block, it is routed as either mill feed or waste. The block size within each of the models was 5 m by 5 m in plan, and 5 m high. The approach used to apply external dilution in an appropriate manner is described in the following sub-sections.

Dilutions skins are considered around a mill feed block if the mill feed block is in contact with neighbouring waste blocks. The first step in this method was to define mill feed and waste blocks. An NSR cut-off value of C$9.96/t was used to define mill feed blocks where only Measured and Indicated blocks would be considered. In these models, there were no Measured blocks. This NSR cut-off value represents the marginal cut-off as it includes the sum of process costs and G&A costs. The second step includes determining the number of waste neighbour contacts for each mill feed block. For each waste neighbour, a dilution tonnage and grade are applied to the mill feed block based on the specified dilution skin thickness. The third step is to reduce the tonnage in the neighbouring waste blocks to remove the diluting material.

The dilution material is added to the parent mill feed block and removed accordingly in neighbouring waste blocks to achieve a material balance. For the whole block method, the diluted model includes revised diluted density and metal grades for all blocks.

SW Model Dilution

The resource model provided for the SW deposit was a whole block, grade model which includes some internal dilution. This means the grade from the wire frames was diluted over the full volume of the block to arrive at a diluted smooth block grade.

The geologic model had been created with grade wireframes prior to assigning the grade into a whole block. AGP believed that this did not adequately reflect the amount of dilution that would be expected with normal mining practice, even with more selective equipment.

AGP also believed that contact dilution would play a role in material sent to the mill. The size of the block in the model was examined as well as the ore zone thicknesses to determine the amount of dilution that would be appropriate for the SW deposit. A dilution skin thickness of 0.4 m was used for SW and resulted in the diluted mill feed containing 9.0% more tonnes and 6.8% lower gold grades than the in-situ mill feed summary. AGP considered this dilution as acceptable due to the narrow mineralization zones being mined.

North Model Dilution

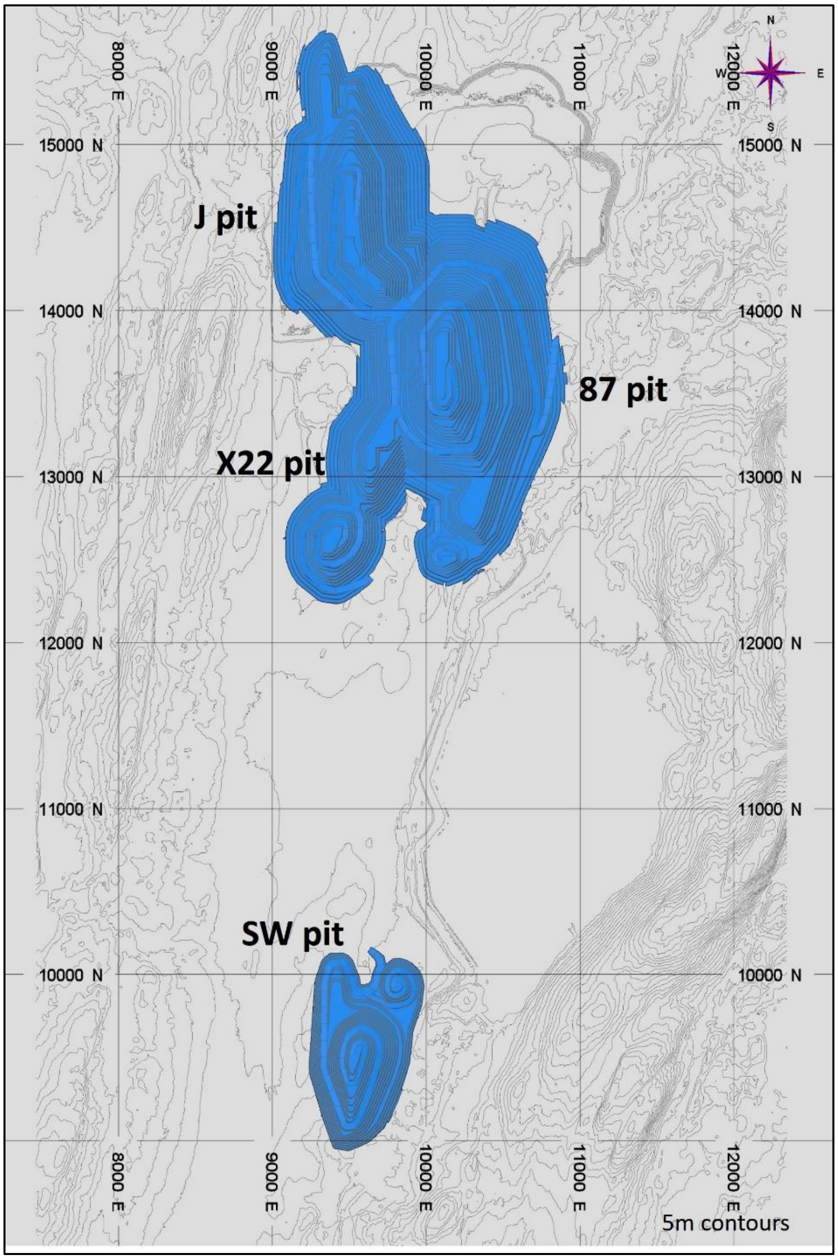
The resource model provided for the north deposit (87, J and X22 zones) was also a whole block, grade model which includes some internal dilution. Similar to the SW model, AGP believed that this did not adequately reflect the amount of dilution that would be expected with normal mining practice, even with more selective equipment.

AGP also believed that contact dilution would play a role in material sent to the mill. The size of the block in the model was examined as well as the ore zone thicknesses to determine the amount of dilution that would be appropriate for the north deposits. A dilution skin thickness of 0.4 m was used for the north deposits and resulted in the diluted mill feed containing 6.2% more tonnes and 4.6% lower gold grades than the in-situ mill feed summary. AGP considered this dilution as acceptable due to the wider mineralization zones being mined.

16.3.4 Pit Design

Pit designs were developed for the J, 87, X22 and SW pit areas. The pit locations are displayed in Figure 16-3. The J pit design consists of two phases of successive pushbacks around the entire pit perimeter. The 87-pit design includes an initial phase 0 at the south end to assist with site water management, followed by phases 1 to 3 in the main portion of the pit. The X22 pit design consists of two phases, with slightly higher grades in the phase 1 at the south. The SW pit design consists of two phases which can be scheduled as satellite phases from the northern pits. The pit optimization shells used to guide the ultimate pits were also used to outline areas of higher value for targeted early mining and phase development. All pits were developed using 10 m bench heights.

Figure 16-3: Pit Locations



Source: AGP (2024)

Geotechnical parameters discussed in Section 16.2 were applied to pit designs as shown in Table 16-7. The parameters agree well with the final slopes of the previously mined J and 87 pits.

Table 16-7: Pit Design Slope Criteria

Pit	Slope	Start Azimuth (degrees)	Material Type	IRA (degrees)	BFA (degrees)	Height Between Berms (m)	Catch Bench Width (m)
J	All slopes		OB, fill	21.8	55	10	18.00
	East	357	Bedrock	47	68	20	10.60
	South	107	Bedrock	55	85	20	12.25
	West	237	Bedrock	52	85	20	13.88
	North	282	Bedrock	57	85	20	11.45
87	All slopes		OB, fill	21.8	55	10	18.00
	East	52	Bedrock	45	65	20	10.70
	South	147	Bedrock	54	85	20	12.65
	West	227	Bedrock	52	85	20	13.85
	Northwest	287	Bedrock	57	85	20	11.15
	Northeast	352	Bedrock	51	85	20	14.65
X22	All slopes		OB, fill	21.8	55	10	18.00
	Northwest	267	Bedrock	55	85	20	12.25
	Southeast	47	Bedrock	47	68	20	10.60
	South & West	177	Bedrock	55	85	20	12.25
SW	All slopes		OB, fill	21.8	55	10	18.00
	East	52	Bedrock	50	68	20	8.50
	South	147	Bedrock	54	70	20	7.50
	West	237	Bedrock	56.5	74	20	7.50
	North	347	Bedrock	52	68	20	7.50

note: 10 m bench heights during mining

Equipment sizing for ramps and working benches is based on the use of 227 tonne rigid frame haul trucks. The operating width used for the truck is 8.6 m. This means that single lane access is 27.8 m (2x operating width plus berm and ditch), and double lane widths are 36.5 m (3x operating width plus berm and ditch). Ramp gradients are 10% for the pits and 8% for the dumps. Working benches were designed for 35 to 40 m minimum mining width on pushbacks.

As the haul road grades exceed 5%, runaway lanes or retardation barriers will need to be incorporated into designs as the project progresses to more detailed studies.

Tonnes and grade for the designed pit phases are reported in Table 16-8 using the diluted tonnes and grade from the models and a mining recovery of 98% to account for additional mill feed losses.

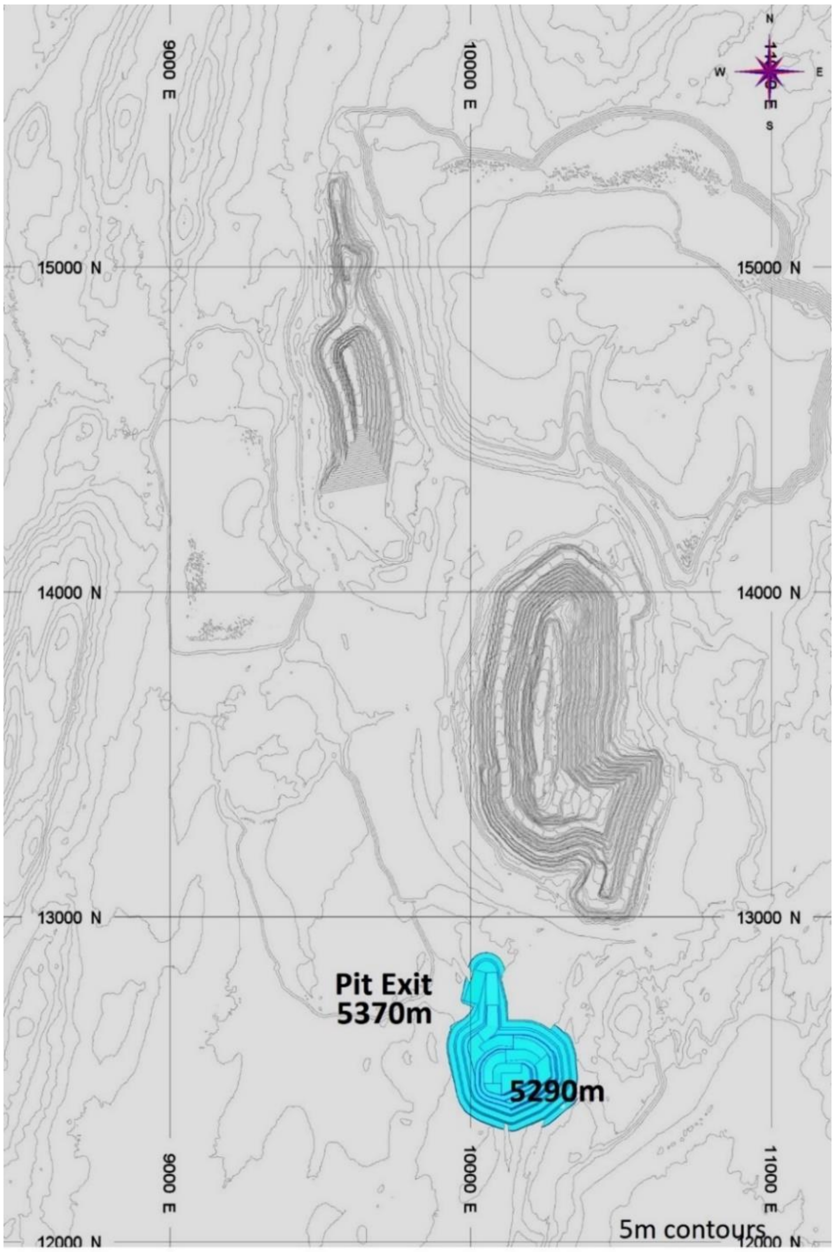
Table 16-8: Pit Phase Tonnages and Grades

Pit	Phase	Ore (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	NSR (C$/t)	Waste (Mt)	Total (Mt)	Strip Ratio
J	1	74.4	0.45	0.06	0.91	29.53	153.0	227.4	2.1
	2	50.8	0.42	0.058	0.84	27.79	164.7	215.5	3.2
J Total		**125.2**	**0.44**	**0.058**	**0.88**	**28.82**	**317.7**	**442.9**	**2.5**
87	0	1.6	0.65	0.04	0.95	42.20	8.5	10.1	5.3
	1	31.6	0.55	0.062	1.17	37.09	139.3	170.9	4.4
	2	69.0	0.58	0.068	1.14	39.38	179.5	248.5	2.6
	3	63.9	0.52	0.055	1.08	34.26	272.0	335.9	4.3
87 Total		**166.1**	**0.55**	**0.062**	**1.12**	**37.00**	**599.4**	**765.5**	**3.6**
X22	1	16.5	0.43	0.07	1.61	29.59	56.5	73.0	3.4
	2	20.0	0.40	0.047	0.79	25.48	53.1	73.0	2.7
X22 Total		**36.4**	**0.41**	**0.058**	**1.16**	**27.34**	**109.6**	**146.0**	**3.0**
SW	1	34.0	0.48	0.05	0.75	29.09	75.1	109.0	2.2
	2	17.9	0.52	0.035	0.78	30.67	69.2	87.1	3.9
SW Total		**51.9**	**0.49**	**0.045**	**0.76**	**29.64**	**144.3**	**196.1**	**2.8**
Troilus Total		**380**	**0.49**	**0.058**	**1.00**	**32.37**	**1,171**	**1,550**	**3.1**

87 Phase 0

87 Phase 0 is the first phase mined in the 87 pit. It has been designed to assist with site water management by allowing storage of excess surface water. This phase is mined from 5380 masl down to 5290 masl. All waste and mill feed accesses will be on the north side of this phase. The design is shown in Figure 16-4.

Figure 16-4: 87 Phase 0 Design

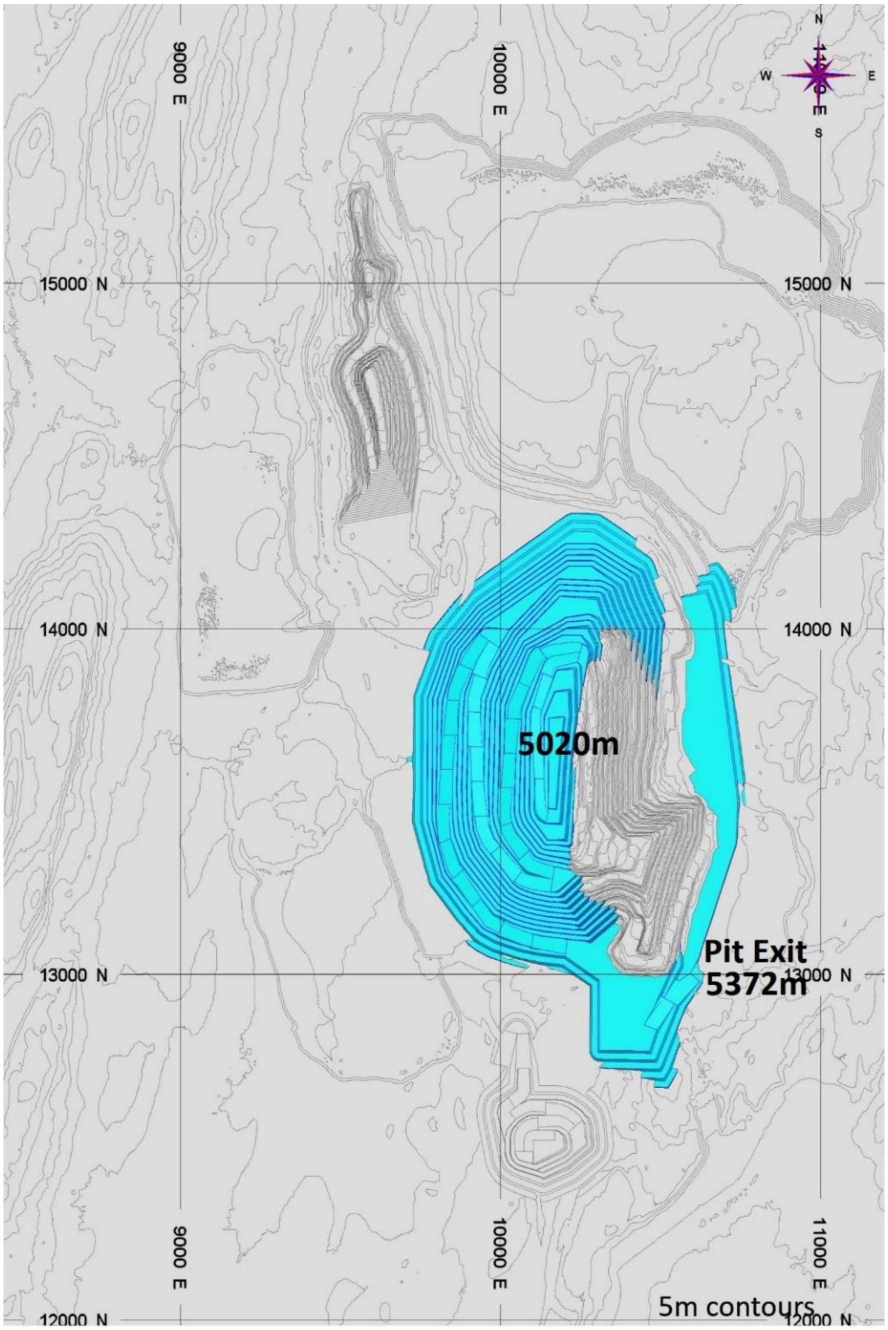


Source: AGP (2024)

87 Phase 1

87 Phase 1 is the first phase mined in the main body of 87 pit. This phase is mined from 5410 masl down to 5020 masl. All waste and mill feed accesses will be on the southeast side of this phase. The design is shown in Figure 16-5.

Figure 16-5: 87 Phase 1 Design

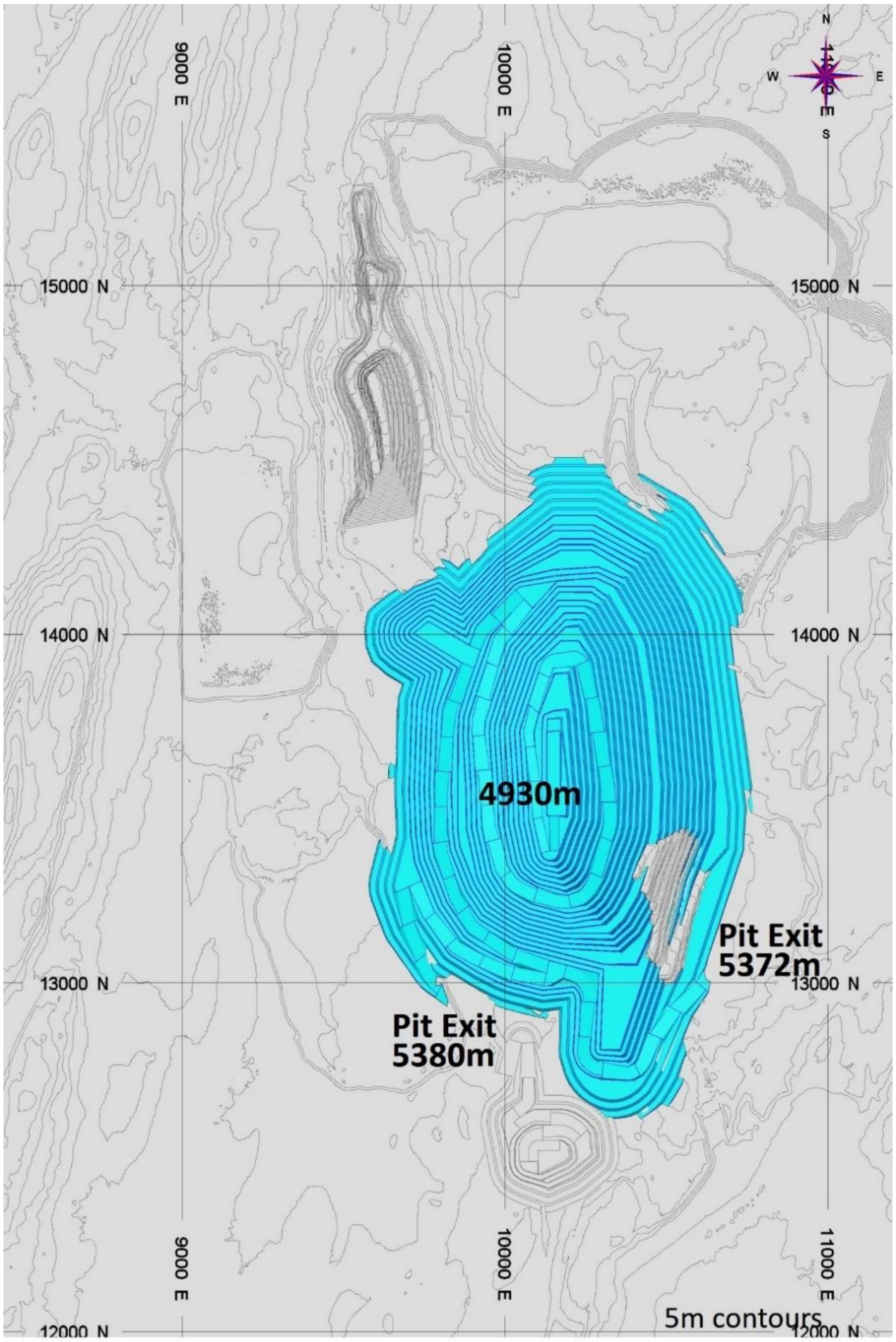


Source: AGP (2024)

87 Phase 2

87 phase 2 is mined from 5410 masl down to 4930 masl. All waste and mill feed accesses will be on the southwest or southeast side of this phase. The design is shown in Figure 16-6.

Figure 16-6: 87 Phase 2 Design

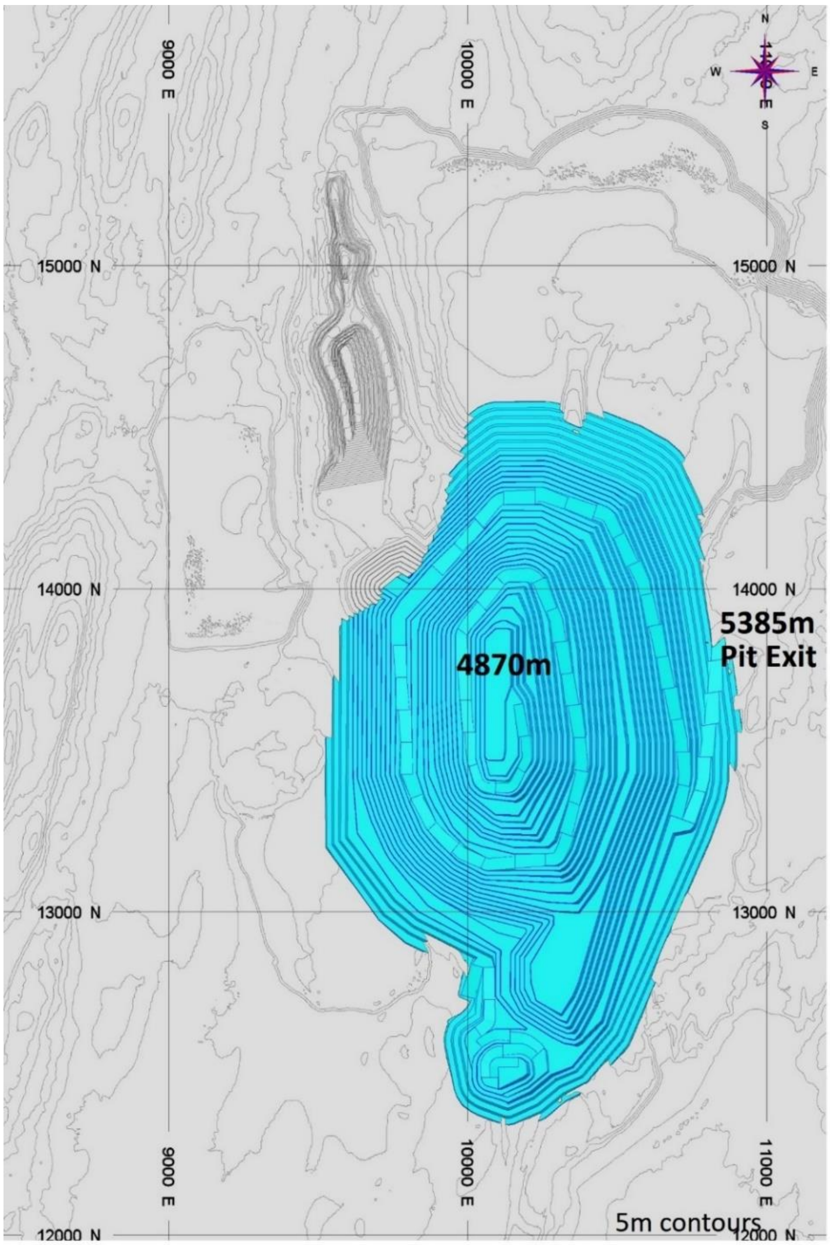


Source: AGP (2024)

87 Phase 3

87 phase 3 is the final phase mined in the 87 pit. This phase is mined from 5420 masl down to 4870 masl. Waste pit exits are available on both the west and east sides of this phase so that there is flexibility in choice of destinations. The west access is available based on advancing the J pit during the schedule. Mill feed material will use the pit exit in the east near the stockpiles. The design is shown in Figure 16-7.

Figure 16-7: 87 Phase 3 Design

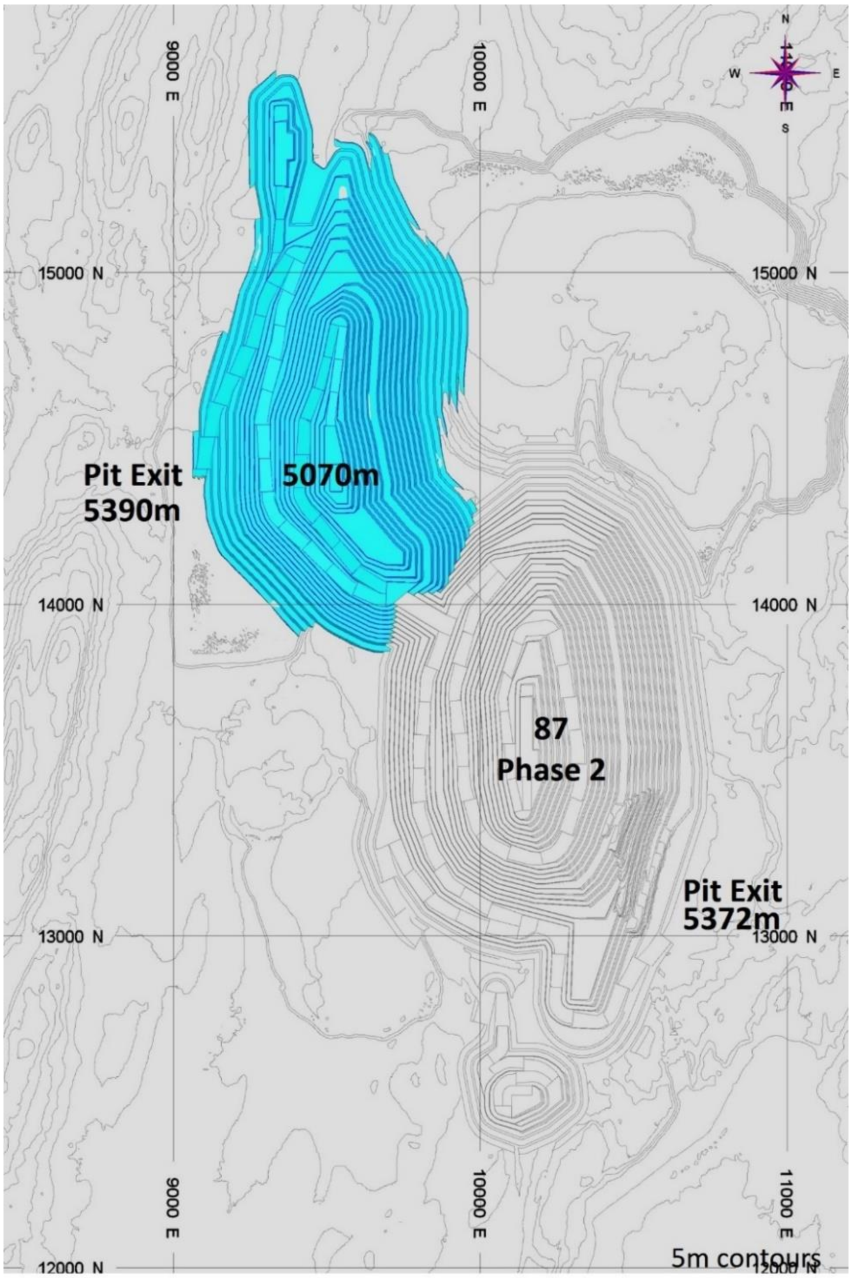


Source: AGP (2024)

J Phase 1

J phase 1 is the first of two phases mined in the J pit. This phase is mined from 5410 masl down to 5170 masl. The mill feed and waste pit exit is located at the west side of the phase. This phase ties into 87 phase 2 at 5200 m elevation, allowing hauls to also exit the 87 pit along the east side. The design is shown in Figure 16-8.

Figure 16-8: J Phase 1 Design

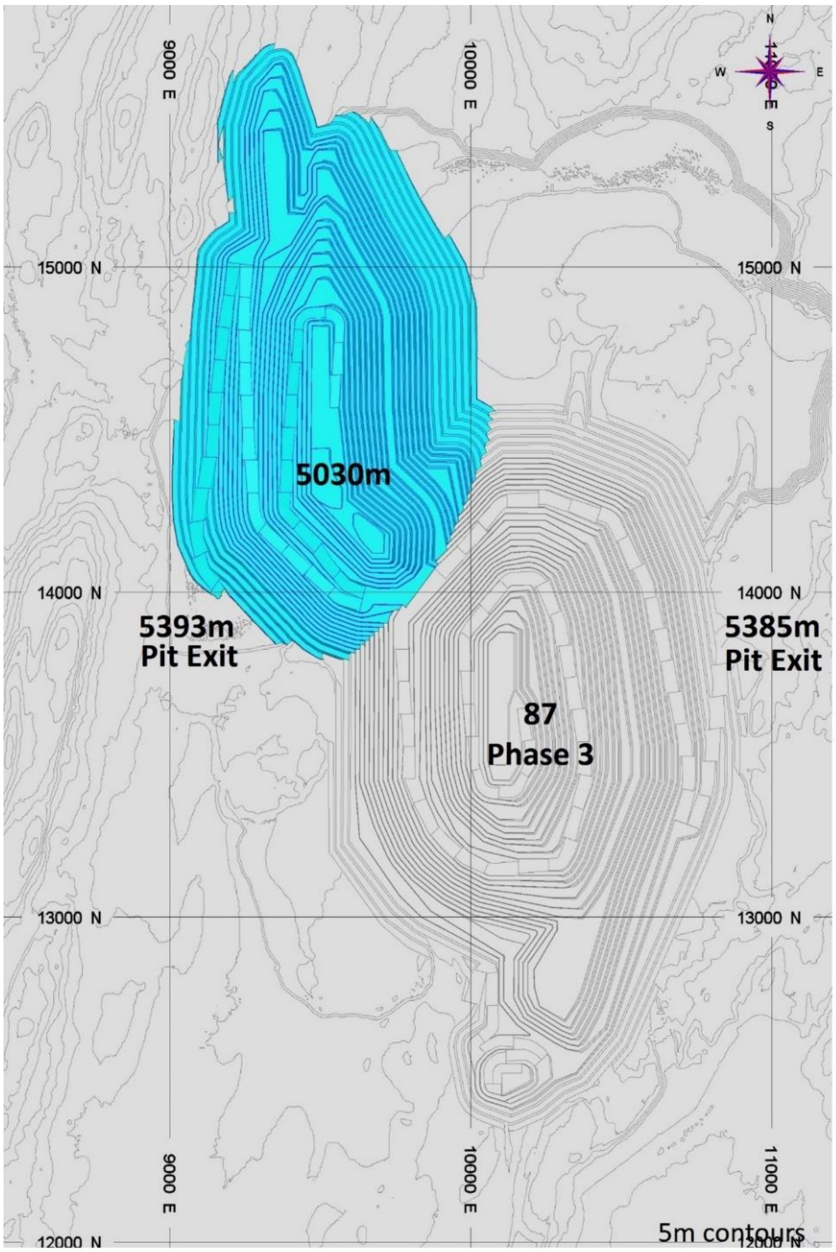


Source: AGP (2024)

J Phase 2

J phase 2 is the final phase mined in the J pit. This phase is mined from 5410 masl down to 5030 masl. The mill feed and waste pit exit are located at the south end of the phase. This phase ties into the 87 phase 3 haul ramp at 5370 m elevation, allowing for shorter hauls to the crusher and stockpiles. This phase required mining of a shallower slopes in historical waste dumps along the southwest and northeast pit walls. The design is shown in Figure 16-9.

Figure 16-9: J Phase 2 Design

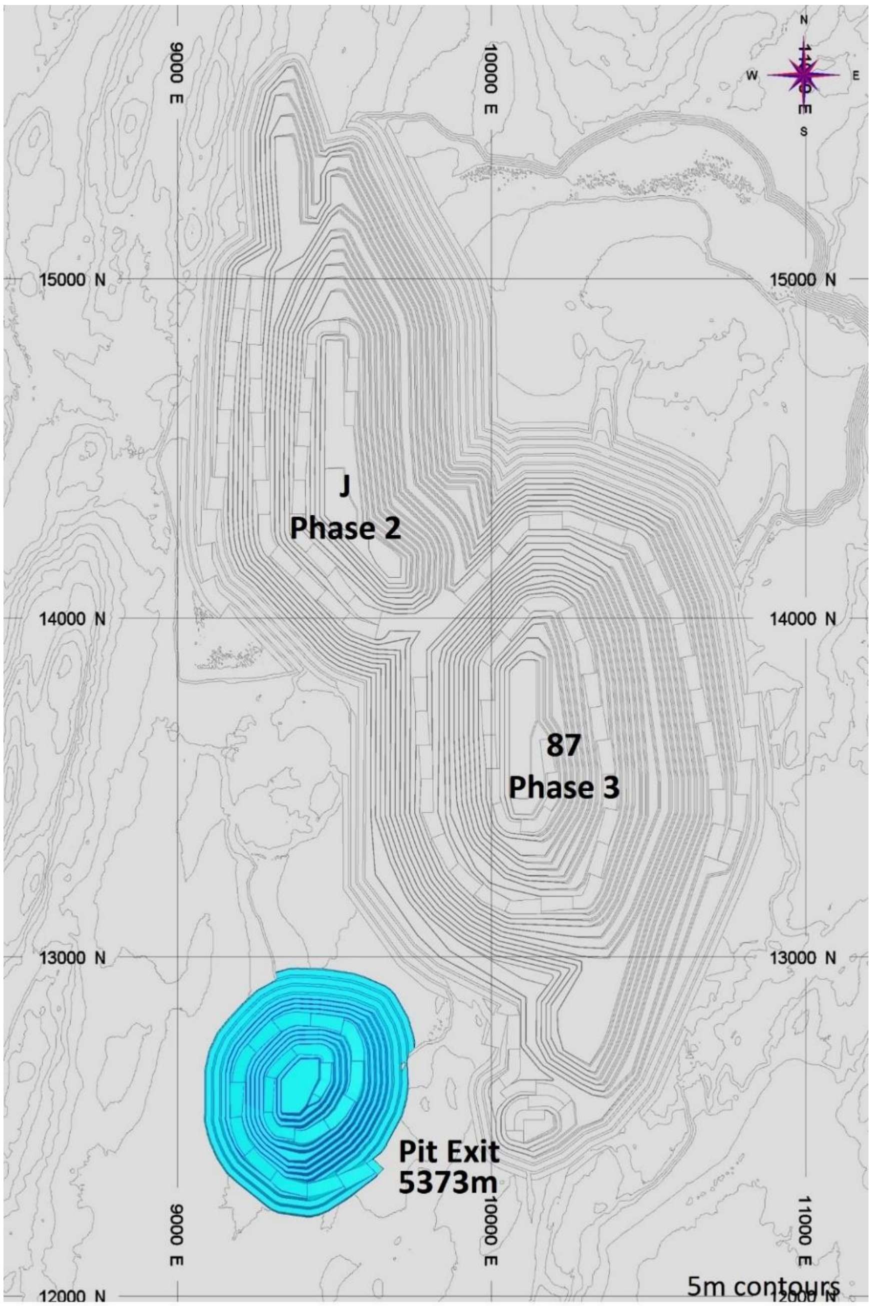


Source: AGP (2024)

X22 Phase 1

X22 phase 1 is the first of two phases mined in the X22 pit. This phase is mined from 5380 masl down to 5170 masl. The mill feed and waste pit exit is located at the east side of the phase. The design is shown in Figure 16-10.

Figure 16-10: X22 Phase 1 Design

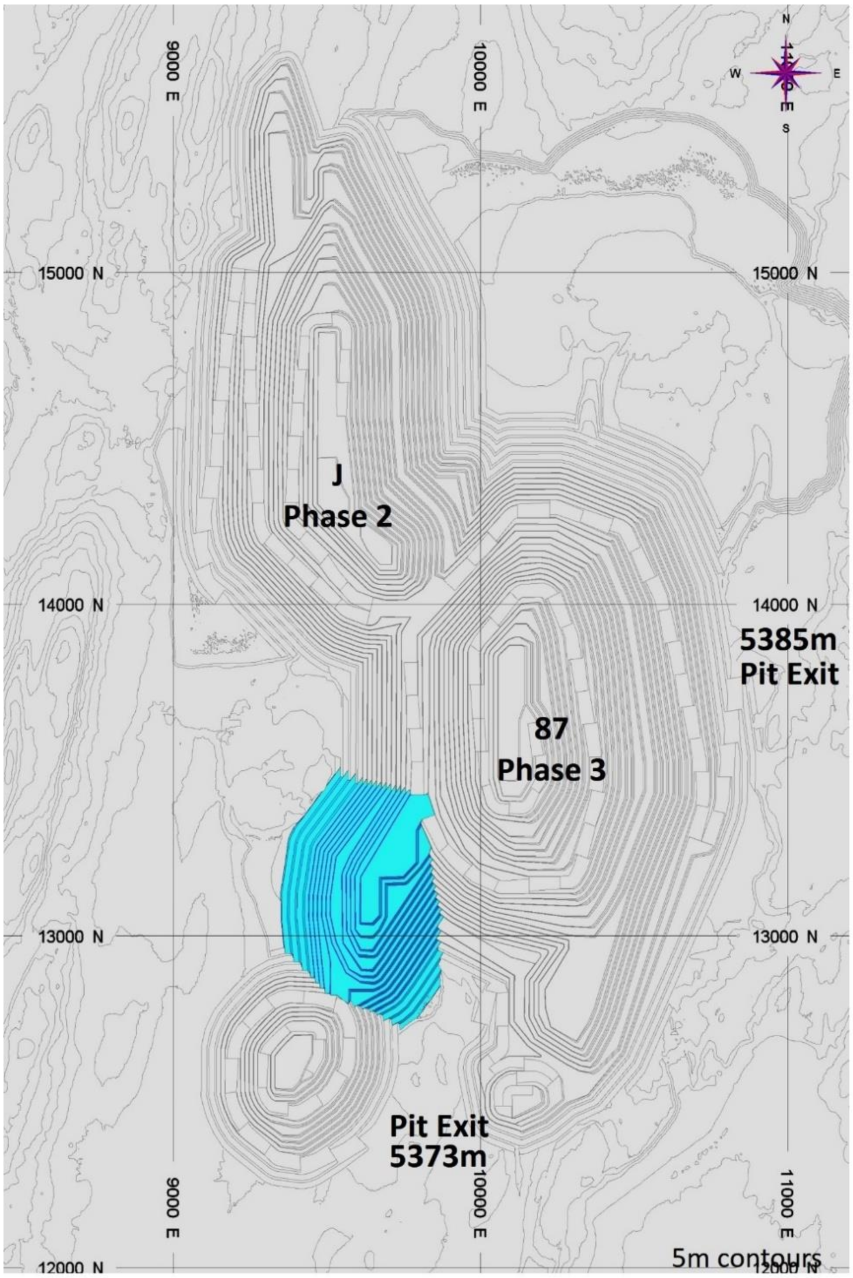


Source: AGP (2024)

X22 Phase 2

X22 phase 2 is the final phase mined in the X22 pit. This phase is mined from 5380 masl down to 5130 masl. The mill feed and waste pit exit are located at the south end of the phase for the upper benches. This phase ties into the 87 phase 3 haul ramp at 5130 m elevation for the lower benches. The final ramp in this phase is mined out after it ties into the 87-haul ramp. The design is shown in Figure 16-11.

Figure 16-11: X22 Phase 2 Design

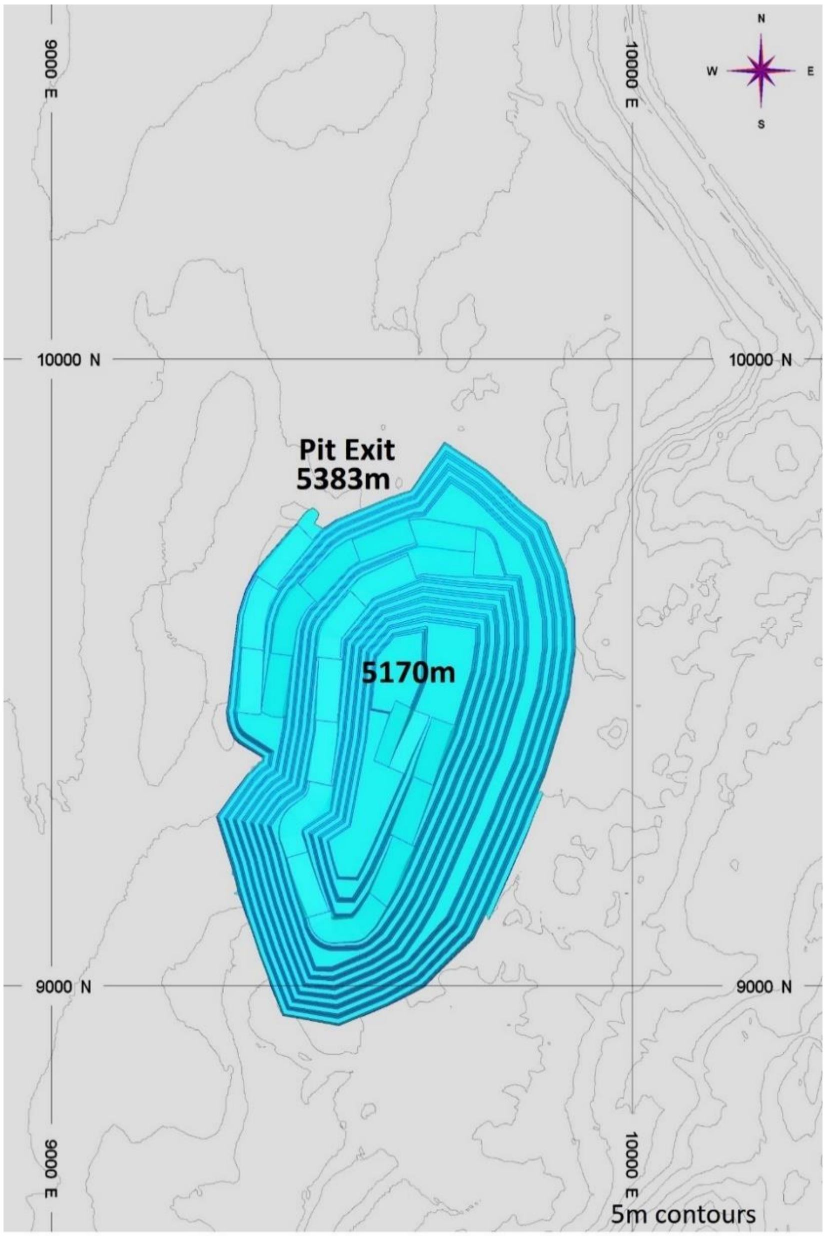


Source: AGP (2024)

SW Phase 1

SW phase 1 is the first phase mined in the SW pit. No previous mining has occurred near the SW deposit. This phase is mined from 5390 masl down to 5170 masl. The mill feed and waste pit exit are located at the north end of the phase, and this works well for the mill location to the north. The design is shown in Figure 16-12.

Figure 16-12: SW Phase 1 Design

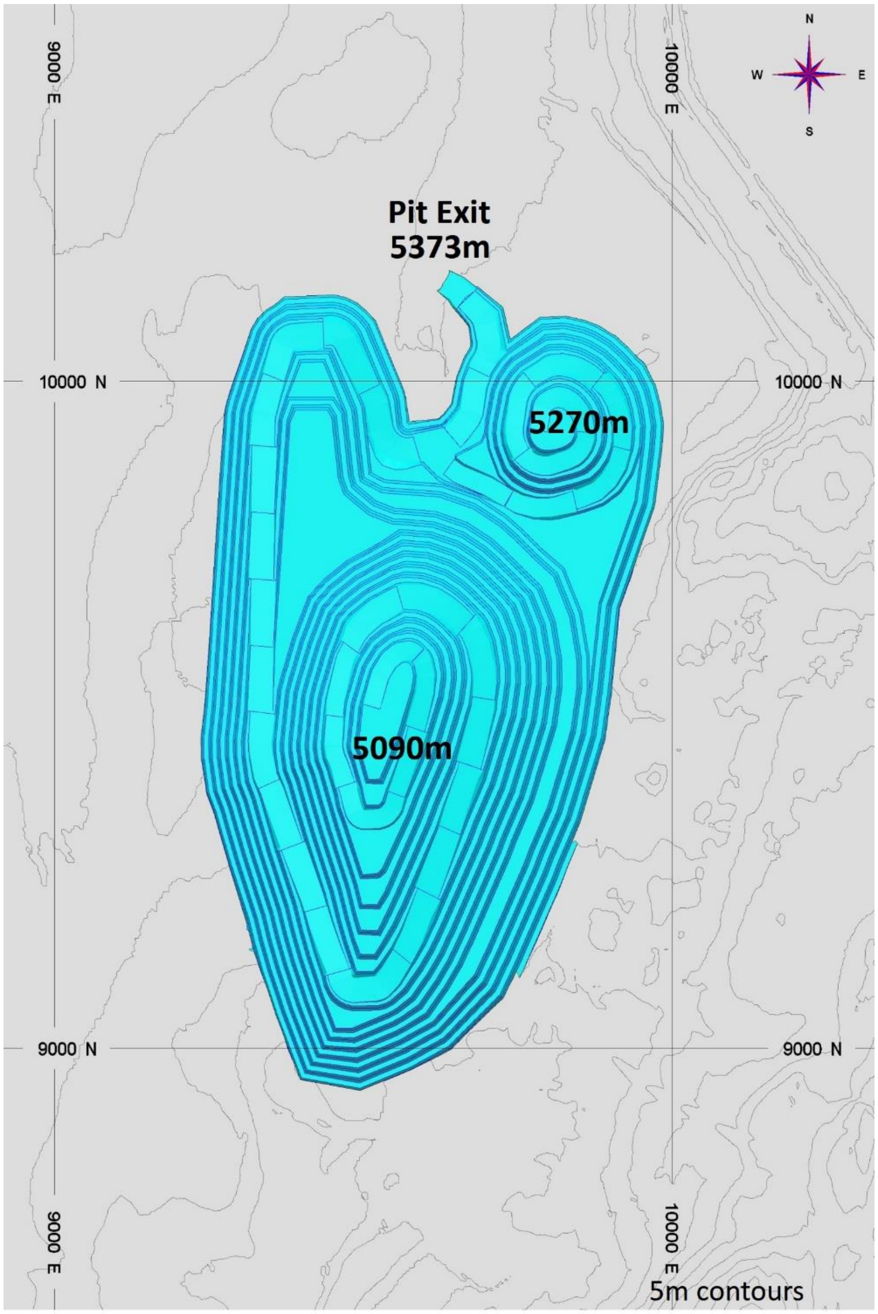


Source: AGP (2024)

SW Phase 2

SW phase 2 is the second and final phase mined in the SW pit. This phase is mined from 5380 masl down to 5090 masl. The pit exit continues to be located at the north end of the phase. The design is shown in Figure 16-13.

Figure 16-13: SW Phase 2 Design



Source: AGP (2024)

16.3.5 Waste Dump Design

Various rock types are present in the material mined within the final pits and have been simplified into overburden and rock types for waste management purposes. Separate overburden storage facilities has been designed for each of the pits, with all other rock material being co-mingled in the remaining waste management facilities. A portion of the historic waste dumps will require re-handling to allow for mining of the expanded J, 87 and X22 pits. This material was treated as rock with a loose density of 2.20 t/m^3 and no drill and blast costs were included in operating costs. The total amount of waste within the mine plan is 1,171 Mt.

The design of the rock waste dumps used a swell factor of 1.30 while a lower swell factor of 1.20 was used for overburden dumps. All waste rock storage facilities were designed with a 37° face slope and overall slopes of 21.8° (2.5H:1V). All overburden storage facilities were designed with a 37° face slope and overall slopes of 18.4° (3H:1V). The overburden facilities were designed with a lift height of 10 m and 16.7 m wide berms. Rock storage facilities were designed with 20 m lift heights and 23.5 m berms.

Waste management facilities will be actively reclaimed as they are developed. Dozers will re-slope them as they are advanced to allow revegetation to occur as soon as possible. Drainage ditches will need to be in place along the waste dump boundaries so that water does not flow directly into other waterways.

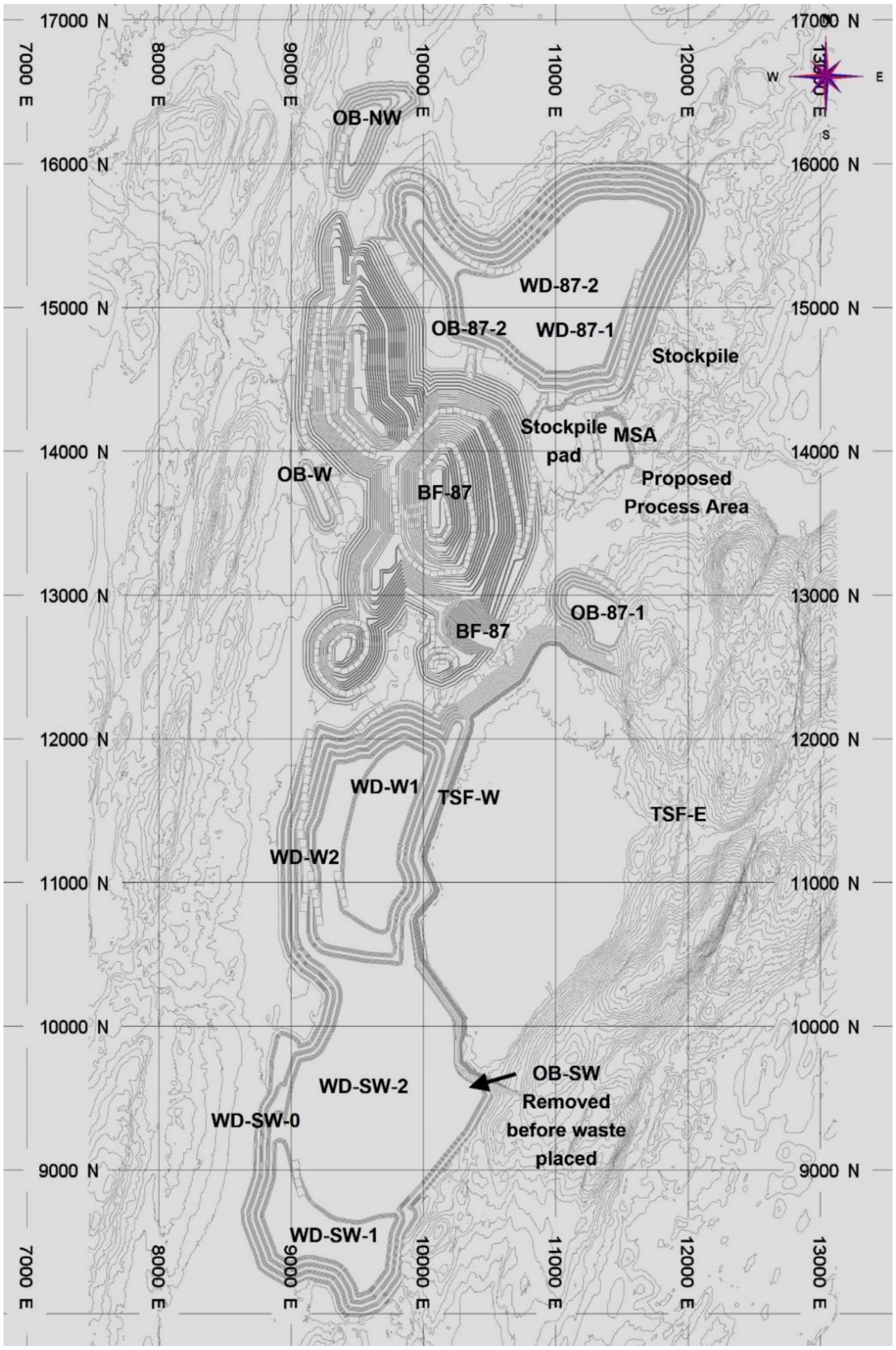
The capacities and top lift elevations for the waste storage facilities are displayed in Table 16-9.

Table 16-9: Waste Storage Facilities Summary

Waste Storage Facility	Capacity	Top Lift Elevation
	(M lcm)	(m)
Stockpile base	1.3	5390
ROM pad	2.9	5417.5
87 Overburden 1	6.4	5440
87 Overburden 2	8.5	5460
87 Waste Dump 1 (East)	38.6	5440
87 Waste Dump 2 (West)	150.4	5500
87 Backfill	7.0	5500
NW Overburden	10.0	5410
West Overburden	1.7	5420
WDW1 Waste Dump	107.0	5440
WDW2 Waste Dump	40.1	5500
X22 backfill	16.4	5370
SW Overburden	6.9	5430
SW Waste Dump 0	25.0	5440
SW Waste Dump 1	39.0	5480
SW Waste Dump 2 (over tails)	100.1	5440
TSF West Embankment	38.5	5440
TSF East Embankment	0.1	5435
Total	**600**	

The waste destinations are displayed in Figure 16-14.

Figure 16-14: Waste Destinations



Source: AGP (2024)

16.3.6 Mine Schedule

The mine schedule for open pit mining consists of 380 Mt of mill feed grading 0.49 g/t gold, 0.058% copper, and 1.0 g/t silver providing mill feed for 22 production years. Open pit waste tonnage totals

1,171 Mt and will be placed into waste storage areas. The overall open pit strip ratio is 3.1:1. The mine schedule utilizes the pit phases described previously to send a maximum of 18.3 Mtpa (50,000 tonnes per day) of feed to the mill facility.

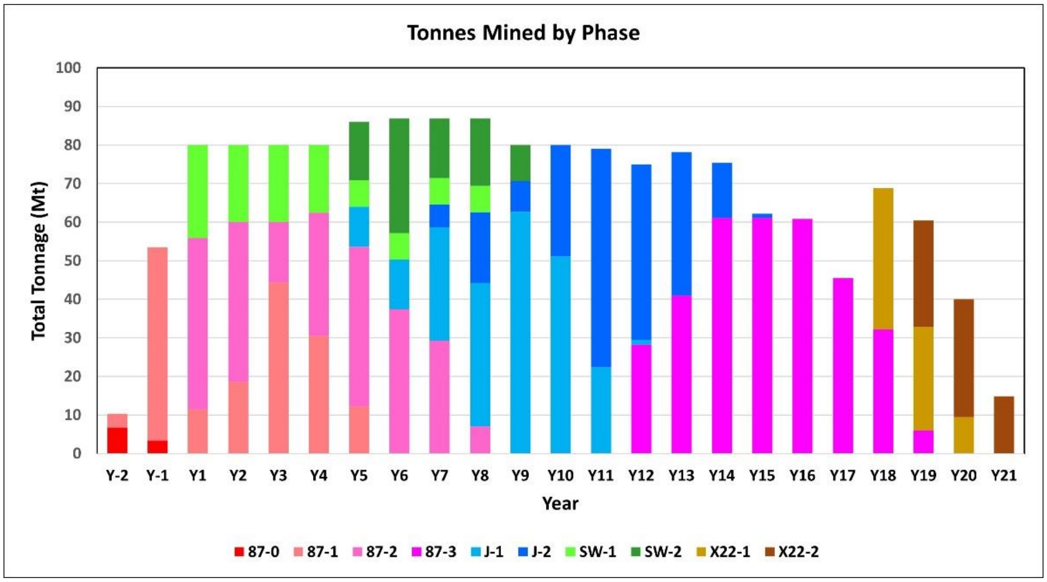
The current mine life includes two years of pre-stripping followed by twenty-one years of mining. Mill feed is stockpiled during the pre-production years, with approximately 0.7 Mt of feed for plant commissioning. A peak stockpile capacity of 48 Mt was reached near the end of year 11. Stockpile material is reclaimed from stockpiles after completion of mining and continues until early into the 22nd year.

The timing of open pit mining total tonnes is displayed in Table 16-10 and Figure 16-15. A maximum descent rate of 9 benches per year per phase was applied for open pit mining to ensure that reasonable mining operations and mill feed control would occur. The open pit mining was starting in year -2 and continued uninterrupted until year 21. Process tonnages and gold grade are shown in Figure 16-16.

Table 16-10: Annual Material Mined by Source

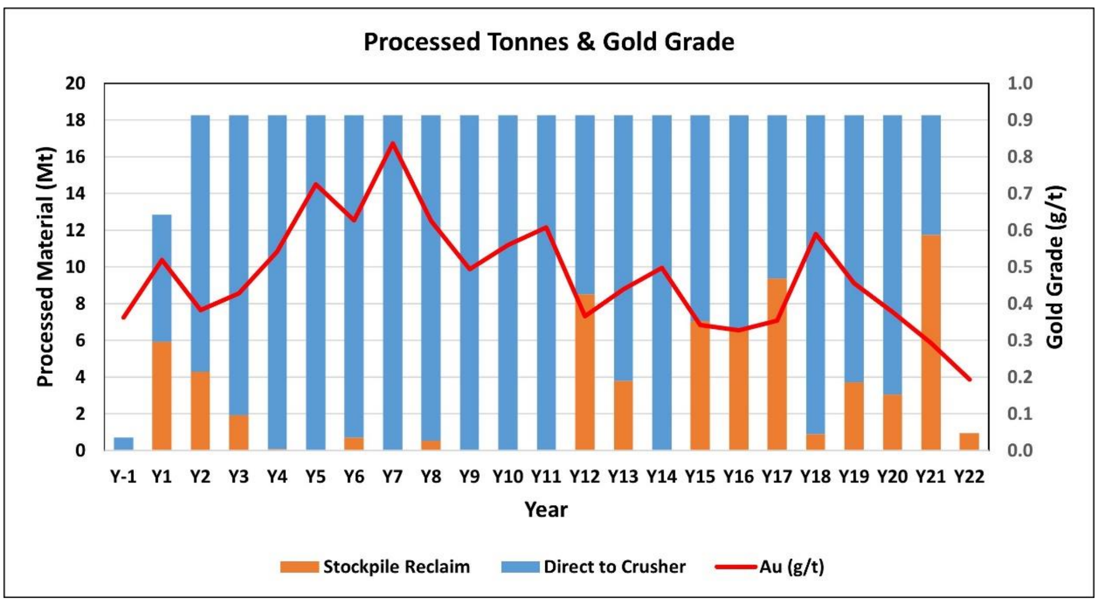
Yr	Mined Material by Source										Total
	87 Pit				J Pit		SW Pit		X22 Pit		
	ph 0 (Mt)	ph 1 (Mt)	ph 2 (Mt)	ph 3 (Mt)	ph 1 (Mt)	ph 2 (Mt)	ph 1 (Mt)	ph 2 (Mt)	ph 1 (Mt)	ph 2 (Mt)	(Mt)
Y-2	7	4	0	0	0	0	0	0	0	0	10
Y-1	3	50	0	0	0	0	0	0	0	0	54
Y1	0	11	44	0	0	0	24	0	0	0	80
Y2	0	19	41	0	0	0	20	0	0	0	80
Y3	0	44	16	0	0	0	20	0	0	0	80
Y4	0	31	32	0	0	0	18	0	0	0	80
Y5	0	12	41	0	10	0	7	15	0	0	86
Y6	0	0	37	0	13	0	7	30	0	0	87
Y7	0	0	29	0	29	6	7	15	0	0	87
Y8	0	0	7	0	37	18	7	17	0	0	87
Y9	0	0	0	0	63	8	0	9	0	0	80
Y10	0	0	0	0	51	29	0	0	0	0	80
Y11	0	0	0	0	22	57	0	0	0	0	79
Y12	0	0	0	28	1	46	0	0	0	0	75
Y13	0	0	0	41	0	37	0	0	0	0	78
Y14	0	0	0	61	0	14	0	0	0	0	75
Y15	0	0	0	61	0	1	0	0	0	0	62
Y16	0	0	0	61	0	0	0	0	0	0	61
Y17	0	0	0	45	0	0	0	0	0	0	45
Y18	0	0	0	32	0	0	0	0	37	0	69
Y19	0	0	0	6	0	0	0	0	27	28	60
Y20	0	0	0	0	0	0	0	0	9	31	40
Y21	0	0	0	0	0	0	0	0	0	15	15
Total	**10**	**171**	**249**	**336**	**227**	**215**	**109**	**87**	**73**	**73**	**1550**

Figure 16-15: Tonnes Mined by Phase



Source: AGP (2024)

Figure 16-16: Process Tonnage and Gold Grade



Source: AGP (2024)

The detailed mine schedule was summarized on an annual basis and is shown in Table 16-11.

Table 16-11: Mine Schedule

	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Mining Summary												
Waste (Mt)	**9.7**	**46.6**	**67.2**	**63.2**	**63.5**	**59.1**	**62.3**	**65.5**	**60.6**	**65.2**	**55.8**	**53.7**
Ore (Mt)	**0.6**	**6.9**	**12.8**	**16.8**	**16.5**	**20.9**	**23.7**	**21.3**	**26.3**	**21.7**	**24.2**	**26.3**
Au (g/t)	0.50	0.45	0.39	0.39	0.45	0.50	0.61	0.56	0.65	0.56	0.42	0.45
Cu (%)	0.04	0.05	0.04	0.04	0.05	0.05	0.07	0.06	0.07	0.07	0.06	0.06
Ag (g/t)	0.70	1.07	0.83	0.85	0.97	1.05	1.10	1.00	1.08	0.97	0.83	0.95
Total (Mt)	**10.3**	**53.5**	**80.0**	**80.0**	**80.0**	**80.0**	**86.0**	**86.9**	**86.9**	**86.9**	**80.0**	**80.0**
Mill Feed (Mt)	**0.0**	**0.7**	**12.9**	**18.2**	**18.3**	**18.3**	**18.3**	**18.3**	**18.3**	**18.3**	**18.3**	**18.3**
Au (g/t)	0.00	0.36	0.52	0.38	0.43	0.54	0.72	0.63	0.84	0.63	0.49	0.56
Cu (%)	0.00	0.06	0.04	0.04	0.05	0.06	0.08	0.06	0.09	0.08	0.06	0.06
Ag (g/t)	0.00	1.23	1.02	0.85	0.93	1.11	1.23	1.07	1.27	1.05	0.90	1.08
Stockpiles (Mt)												
LG: 9.66<NSR<14	0.2	1.7	4.0	5.2	3.5	5.5	9.4	13.2	17.5	21.5	26.8	31.4
MG1: 14<NSR<18	0.1	1.2	2.7	0.0	0.0	0.6	2.2	1.5	5.0	4.6	5.3	8.6
MG2: 18<NSR<22	0.1	0.9	0.0	0.0	0.0	0.0	0.0	0.0	0.1	0.0	0.0	0.0
HG: NSR>22	0.3	3.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Total (Mt)	**0.6**	**6.8**	**6.7**	**5.2**	**3.5**	**6.1**	**11.5**	**14.6**	**22.7**	**26.1**	**32.0**	**40.1**
Reclaim (Mt)	**0.0**	**0.0**	**5.9**	**4.3**	**1.9**	**0.1**	**0.0**	**0.7**	**0.0**	**0.5**	**0.0**	**0.0**
Material Mvmt (Mt)	**10.3**	**53.5**	**85.9**	**84.3**	**81.9**	**80.1**	**86.0**	**87.6**	**86.9**	**87.4**	**80.0**	**80.0**

	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20	Y21	Y22	Total
Mining Summary													
Waste (Mt)	**52.9**	**62.0**	**60.2**	**56.1**	**50.9**	**49.2**	**36.6**	**51.5**	**45.9**	**24.8**	**8.4**	**0.0**	**1,171**
Ore (Mt)	**26.1**	**13.0**	**17.9**	**19.3**	**11.2**	**11.6**	**8.9**	**17.4**	**14.5**	**15.2**	**6.5**	**0.0**	**380**
Au (g/t)	0.49	0.40	0.43	0.48	0.43	0.40	0.52	0.61	0.52	0.41	0.47	0.00	0.49
Cu (%)	0.06	0.06	0.05	0.05	0.03	0.04	0.07	0.07	0.07	0.06	0.05	0.00	0.06
Ag (g/t)	0.89	0.80	0.87	1.01	0.86	0.90	1.26	1.31	1.28	1.27	0.74	0.00	1.00
Total (Mt)	**79.0**	**75.0**	**78.2**	**75.4**	**62.1**	**60.8**	**45.5**	**68.8**	**60.4**	**40.0**	**14.9**	**0.0**	**1,550**
Mill Feed (Mt)	**18.3**	**18.3**	**18.3**	**18.3**	**18.3**	**18.3**	**18.3**	**18.3**	**18.3**	**18.3**	**18.3**	**0.9**	**380**
Au (g/t)	0.61	0.37	0.44	0.50	0.34	0.33	0.35	0.59	0.46	0.38	0.29	0.19	0.49

Cu (%)	0.06	0.06	0.05	0.05	0.03	0.04	0.05	0.07	0.06	0.06	0.04	0.04	0.06
Ag (g/t)	1.01	0.80	0.89	1.03	0.76	0.79	0.92	1.27	1.14	1.16	0.65	0.60	1.00
Stockpiles (Mt)													
LG: 9.66<NSR<14	35.6	38.9	42.3	43.4	36.4	29.7	20.4	19.5	15.7	12.7	0.9	0.0	
MG1: 14<NSR<18	12.3	3.8	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	
MG2: 18<NSR<22	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	
HG: NSR>22	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	
Total (Mt)	**47.9**	**42.7**	**42.3**	**43.4**	**36.4**	**29.7**	**20.4**	**19.5**	**15.7**	**12.7**	**0.9**	**0.0**	
Reclaim (Mt)	**0.0**	**8.5**	**3.8**	**0.0**	**7.1**	**6.6**	**9.4**	**0.9**	**3.7**	**3.1**	**11.7**	**0.9**	**69.2**
Material Mvmt (Mt)	**79.0**	**83.5**	**82.0**	**75.4**	**69.2**	**67.4**	**54.8**	**69.7**	**64.1**	**43.1**	**26.6**	**0.9**	**1,620**

Mineralized material in the production schedule was split into four NSR grade bins to assist in realistic mill feed grade management. The stockpiled material, together with pit phase sequencing, was utilized to ensure mill feed is processing the best material available during the schedule. Table 16-12 displays a summary of the reserve classifications for the mill feed by pit. There are no proven reserves.

Table 16-12: Reserve Summary of Scheduled Material

Pit	Probable Reserves					
	Mill Feed (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	AuEq (g/t)	CuEq (%)
87	166	0.55	0.062	1.12	0.66	**0.43**
J	125	0.44	0.058	0.88	0.54	**0.36**
X22	36	0.41	0.058	1.16	0.52	**0.34**
SW	52	0.49	0.045	0.76	0.58	**0.35**
Total	**380**	**0.49**	**0.058**	**1.00**	**0.59**	**0.39**

Note: This mineral reserve estimate has an effective date of January 15, 2024, and is based on the mineral resource estimate dated October 2, 2023, for Troilus Gold by AGP Mining Consultants Inc. The mineral reserve estimate was completed under the supervision of Willie Hamilton, P.Eng. of AGP, who is a Qualified Person as defined under NI 43-101. Mineral Reserves are stated within the final pit designs based on a US$1,550/oz gold price, US$20.00/oz silver price and US$3.50/lb copper price. An NSR cut-off of C$9.96/t was used to define reserves. The life-of-mine mining cost averaged C$3.99/t mined, preliminary processing costs were C$8.02/t ore and G&A was C$1.94/t ore placed. The metallurgical recoveries were varied according to gold head grade and concentrate grades. 87 pit recoveries for equivalent grades were 95.5%, 94.7% and 98.2% for gold, copper, and silver, respectively. J pit recoveries for equivalent grades were 93.1%, 89.3% and 88.9% for gold, copper, and silver, respectively. X22 pit recoveries for equivalent grades were 95.5%, 94.7% and 98.2% for gold, copper, and silver, respectively. SW pit recoveries for equivalent grades were 85.7%, 91.5% and 85.6% for gold, copper, and silver, respectively. The formulas used to calculate equivalent values are as follows, for 87 Pit AuEq = Au + 1.5361*Cu +0.0133 *Ag, for J Pit AuEq = Au + 1.4849*Cu +0.0123 *Ag, for SW Pit AuEq = Au + 1.6535*Cu +0.0129 *Ag, for X22 Pit AuEq = Au + 1.5361*Cu +0.0133 *Ag.

Two preproduction years have been defined for getting the project ready for full mill production. Years -2 and -1 has mining initiated in phases 0 and 1 of the 87 pit. Phase 0 is mined complete in year -1 so that it may can provide water storage for surface water management activities. During preproduction, a total of 64 Mt of material will be moved as the project ramps up. The mill feed stockpile pad is established so that mill feed material can be sent to the mill stockpile. Waste is also used to create a pad for the crusher and mine maintenance facilities. 7.5 Mt of mill feed ore is mined with only 0.7 Mt of low-grade material being fed to the mill in Year -1 to complete mill commissioning. 6.8 Mt of mill feed material remain in stockpiles at the end of preproduction in anticipation of plant ramp-up to full production. Significant activities near the pit will include establishing proper roads to the mill feed crusher and to the various waste storage areas. Operationally, ditching around the pits to intercept surface run-off will help to minimize reductions in mine production. A diversion ditch is completed by contractors during pre-production along the west side of the project. Overburden material is directed to dedicated storage areas to the north of the existing TSF as well as on top of the historic 87 waste rock facility. Waste rock is initiated for TSF lifts in year -1 and continues until year 8 to ensure adequate tailings storage capacity is available for mill production.

Year 1 production assumes the plant will be able to ramp up and mill 12.8 Mt of mill feed. The last 3 months of the year were scheduled at the 50 ktpd nameplate capacity in the mill. Mill feed will be from stockpile and both the 87 and SW pits. 87 phase 2 and SW phase 1 are started in year 1 and remain as the only active mining areas along with 87 phase 1 until the end of year 4. During these 4 years, 87 pit waste is sent to northeast of the historic 87 waste rock facility, TSF lifts or the waste rock facility to the west of the TSF. SW waste is sent to a waste rock facility to the west of the pit.

Year 5 mining is noted for the completion of 87 phase 1 and the start of J phase 1 and SW phase 2. Overburden from J pit is sent the north, and its waste is split to the west waste facility and 87 waste rock facility. Years 5 to 8 have the highest mining rates in the mine schedule at approximately 87 Mtpa of ore and waste. In year 7, J phase 2 is started. In year 8, 87 phase 2 and SW phase 1 are mined complete.

Year 9 is the final year of mining in the SW pit. After completion, this pit becomes available for tailings storage as the TSF is expected to reach current design capacity during year 10.

Years 10 and 11 have mining exclusively from the two phases in J pit. J phase 1 is mined complete in year 12 and this coincides with the start of 87 phase 3 mining.

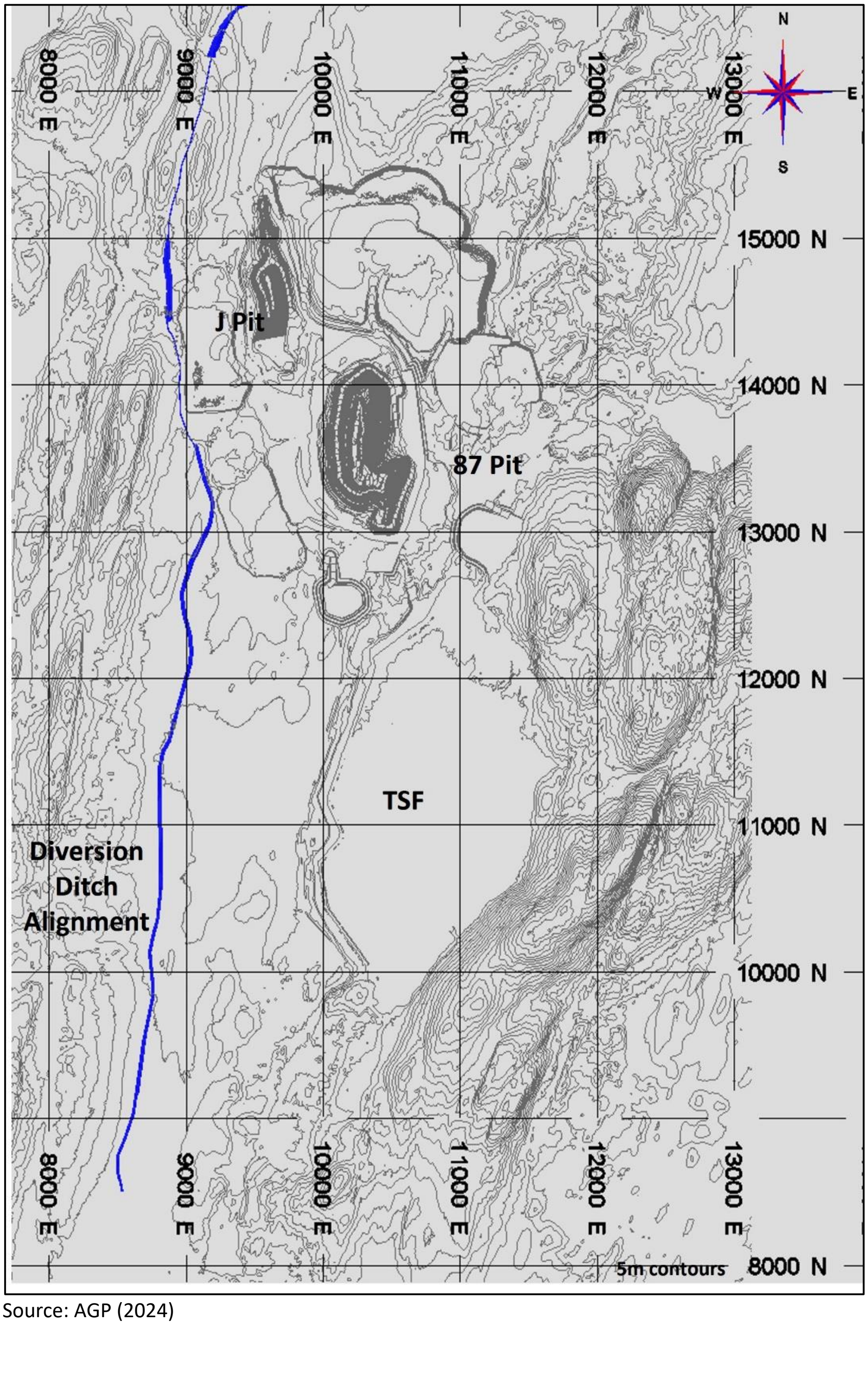
Year 13 to 15 has mining activities in 87 phase 3 and J phase 2. J phase 2 is completed during year 15 and becomes available for tailings storage. The tailing storage into SW pit will transition to J pit as required. Once tailings are no longer being deposited into the SW pit, a waste rock facility will be established to cover the tailings in the SW pit and tie into the surrounding storage facilities. This SW waste facility will be used for extra storage capacity between years 15 to 19. Approximately 14 Mt of waste will also be backfilled into the 87 pit from the pit edge near the TSF.

Year 16 and 17 has mining in 87 phase 3 with tailings being directed into J pit. 87 phase 3 is expected to be mined complete early in year 19.

X22 phases 1 and 2 will begin mining in years 18 and 19, respectively. Year 21 is the final year of pit mining. The 8 Mt of waste from year 21 is sent to 87 as backfill but could alternatively be sent to other waste facilities or as backfill into X22 phase 1. Tailings storage is expected to reach the J pit storage capacity in early year 19, so will transition to deposition into the mined out 87 pit afterwards. The remainder of year 21 and until early in year 22 has the mill being fed by reclaimed material from stockpiles.

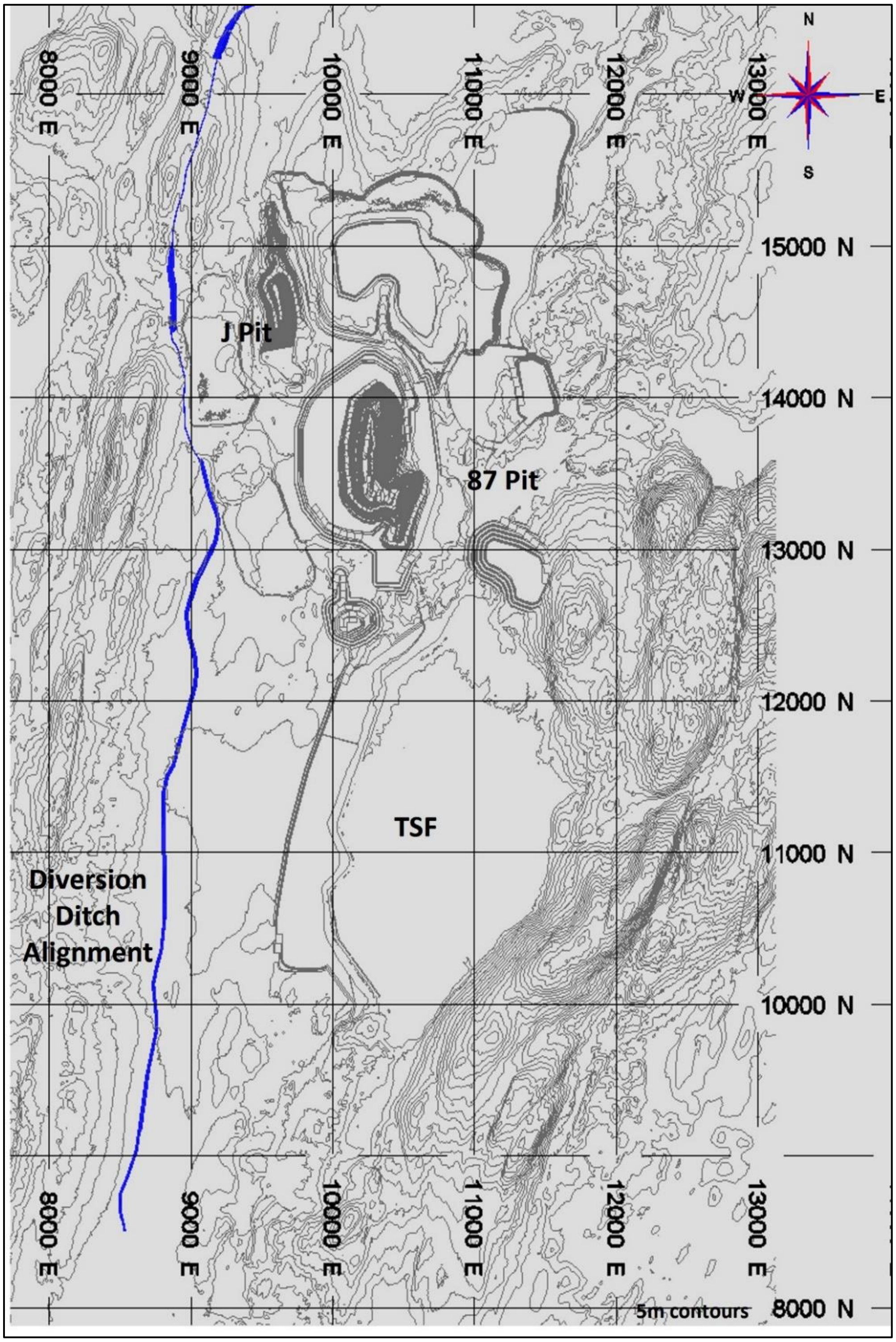
Selected end-of-year positions for the open pits and waste storage facilities are shown in Figure 16-17 to Figure 16-26.

Figure 16-17: End of Year -2



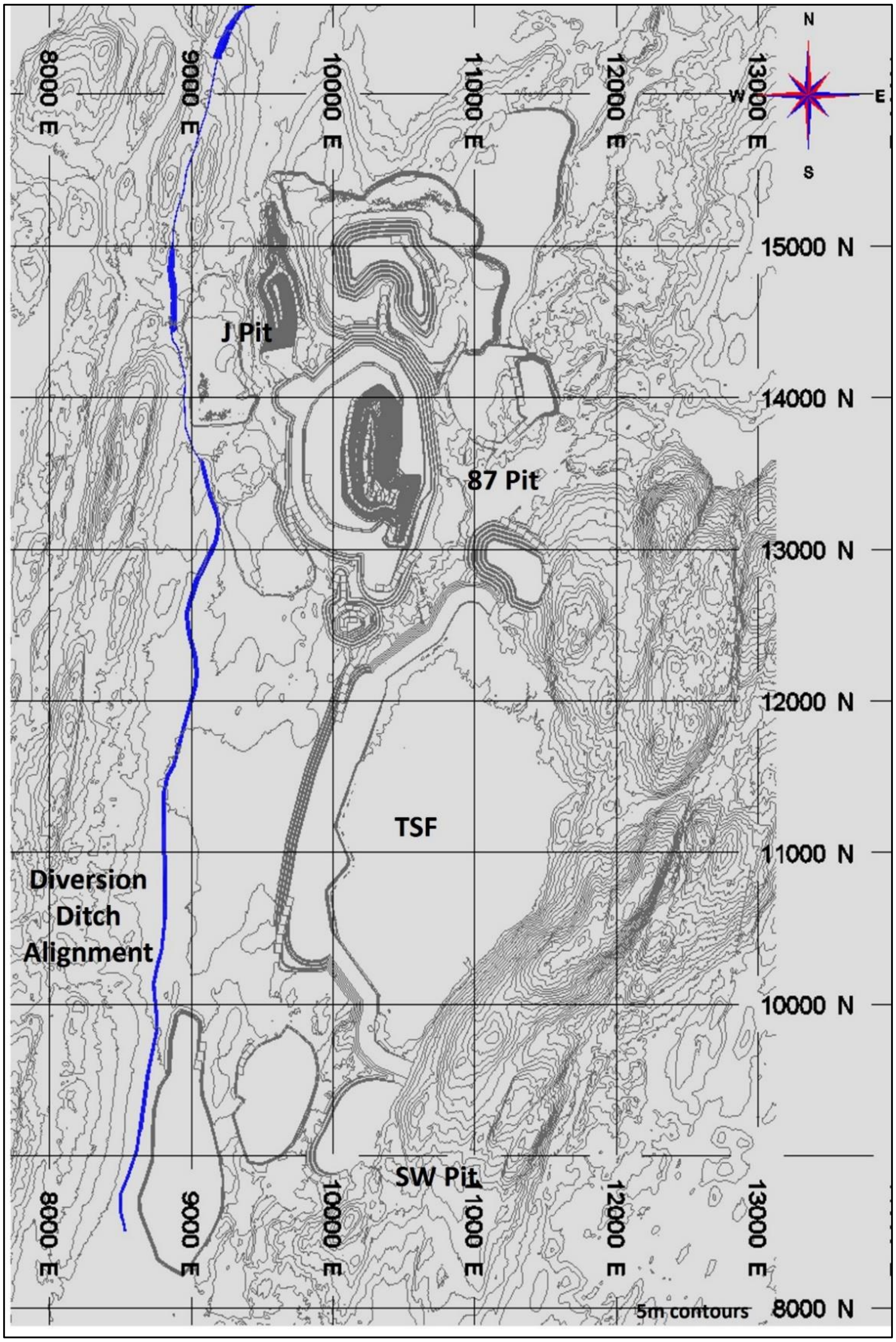
Source: AGP (2024)

Figure 16-18: End of Year -1



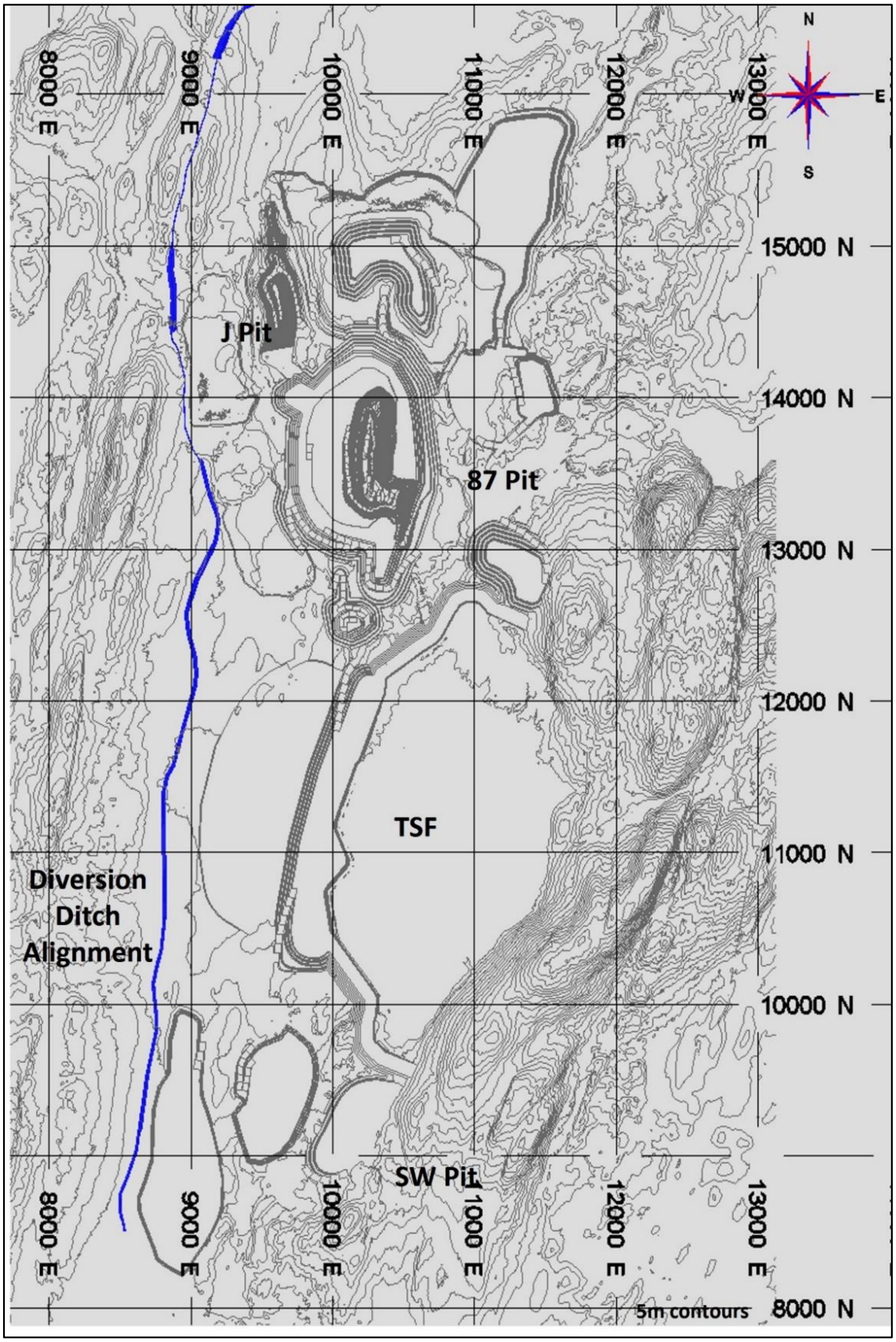
Source: AGP (2024)

Figure 16-19: End of Year 1



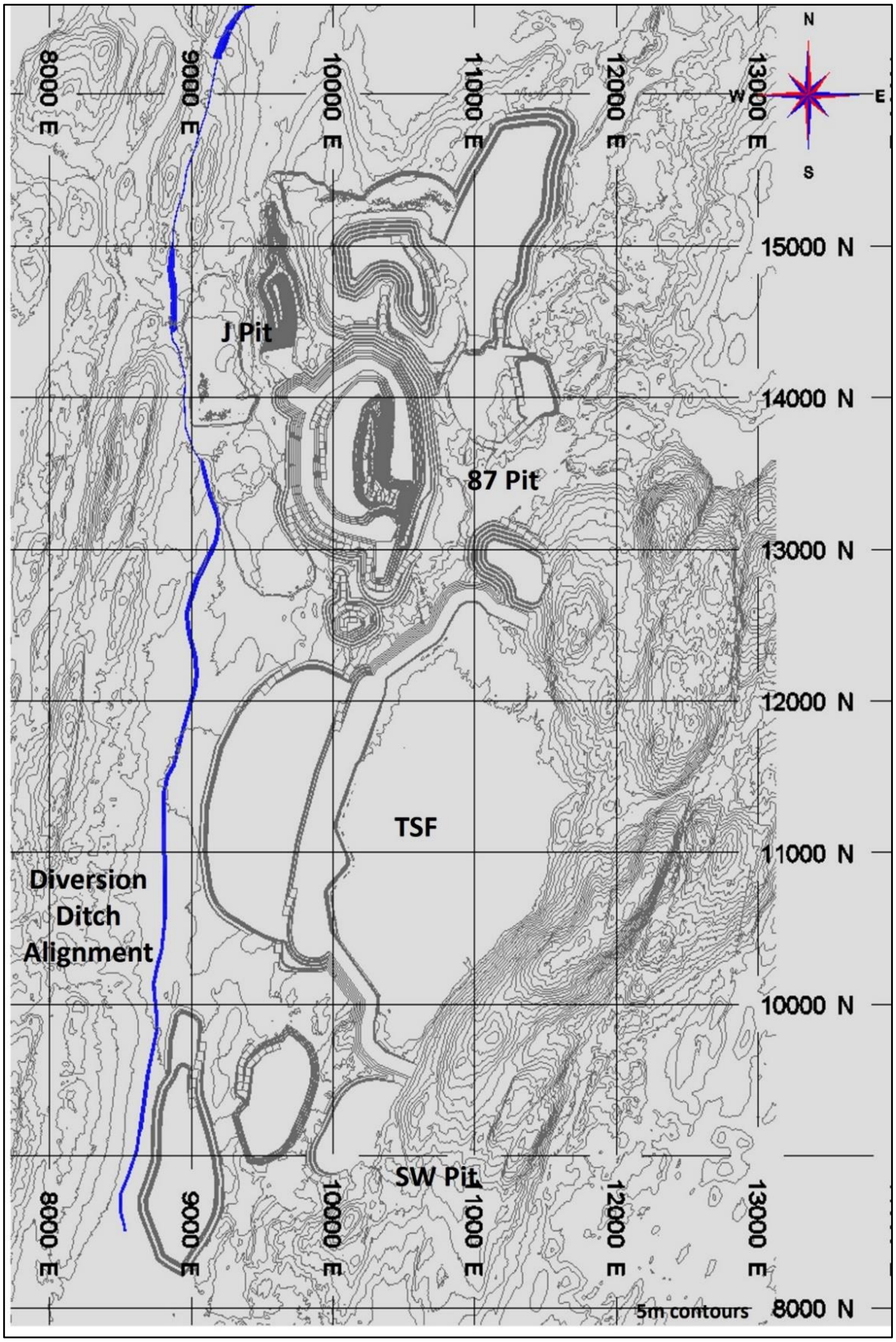
Source: AGP (2024)

Figure 16-20: End of Year 2



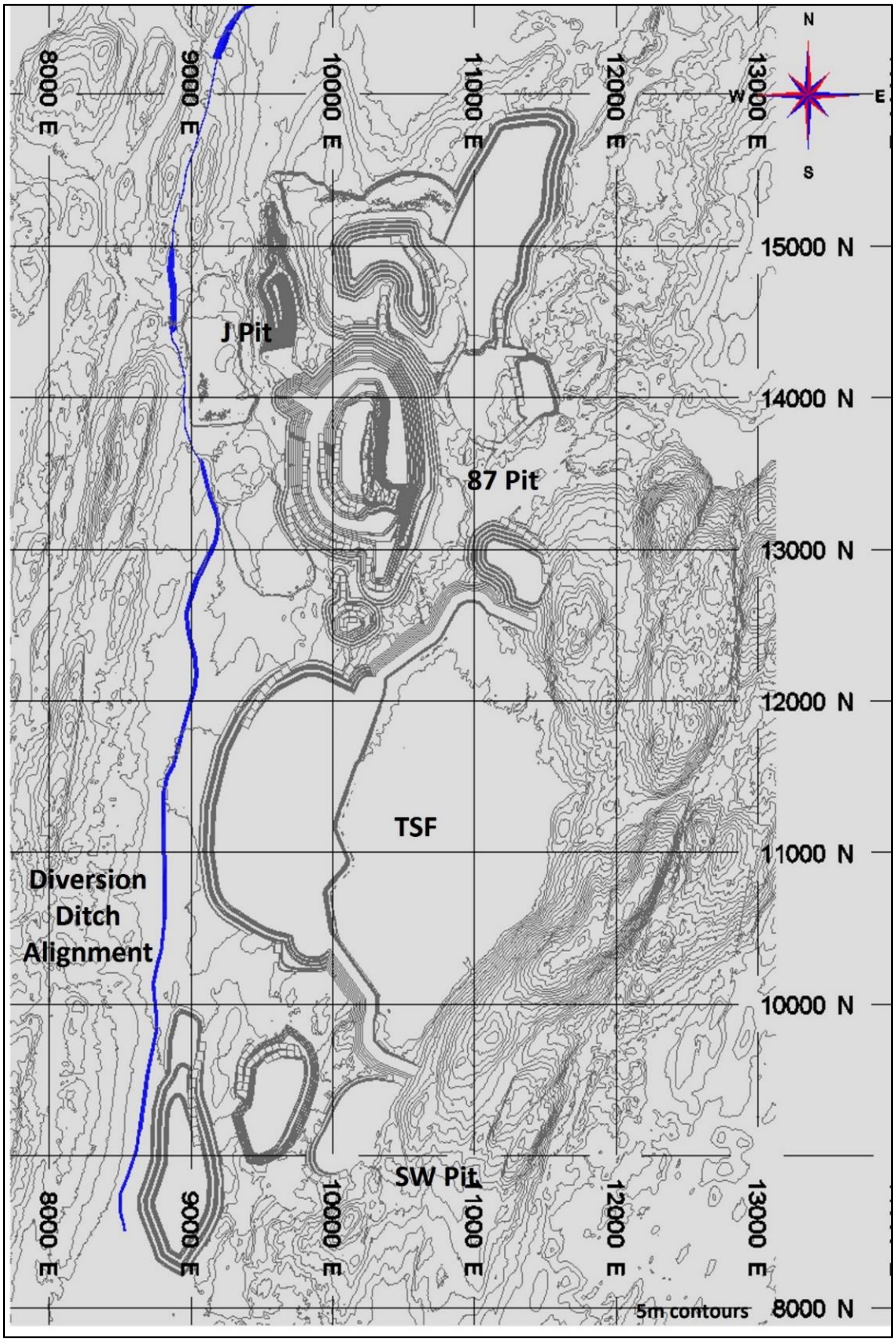
Source: AGP (2024)

Figure 16-21: End of Year 3



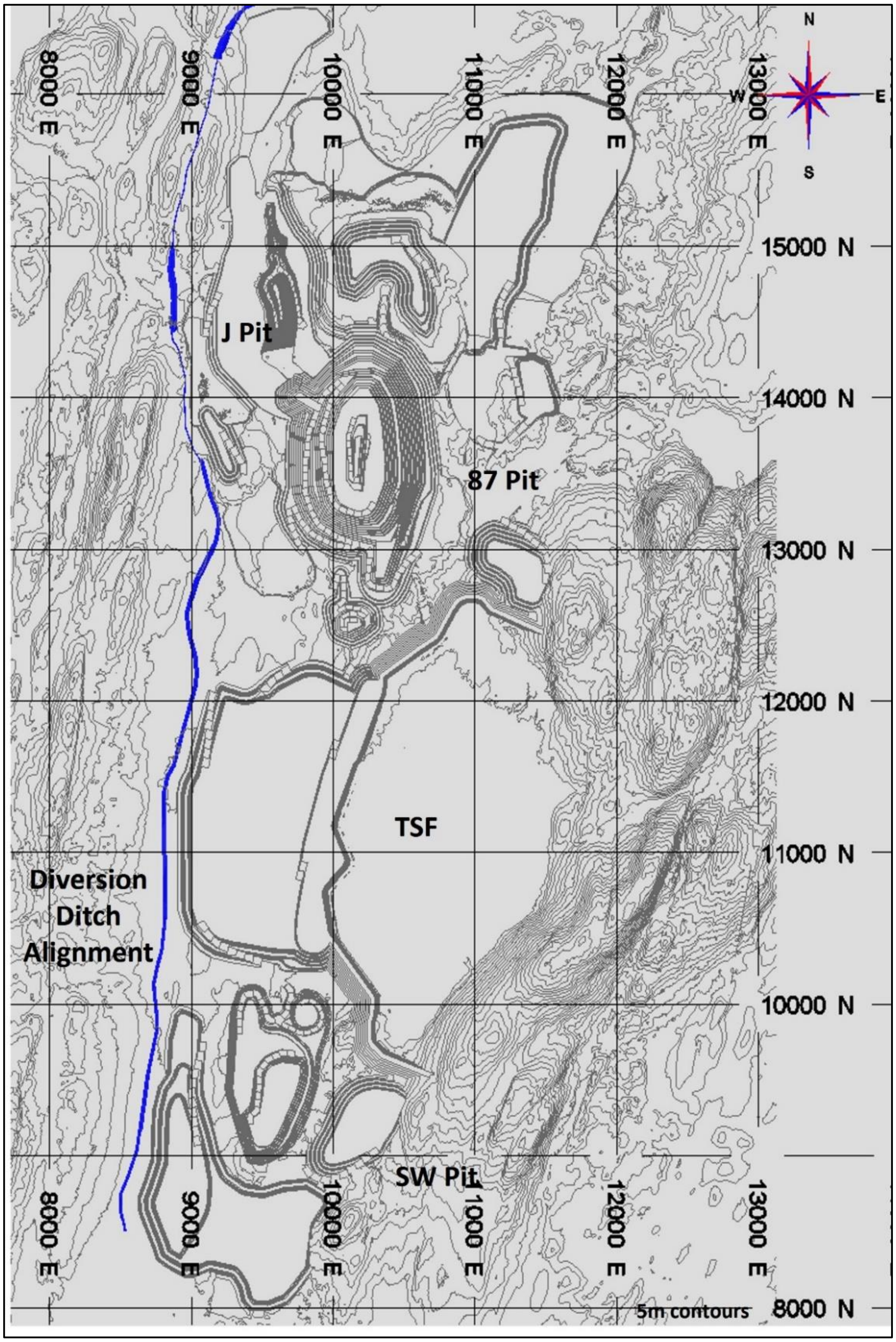
Source: AGP (2024)

Figure 16-22: End of Year 4



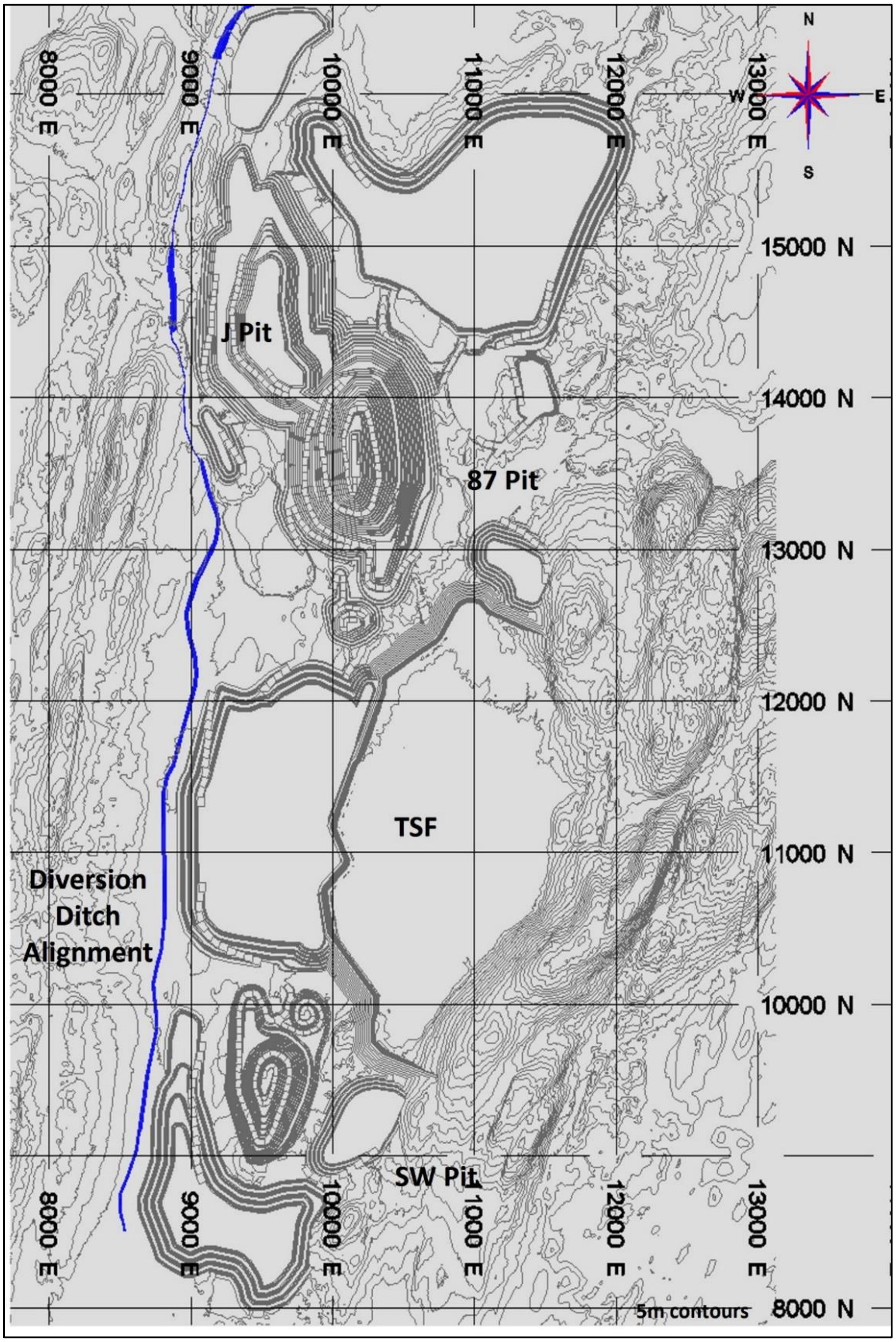
Source: AGP (2024)

Figure 16-23: End of Year 5



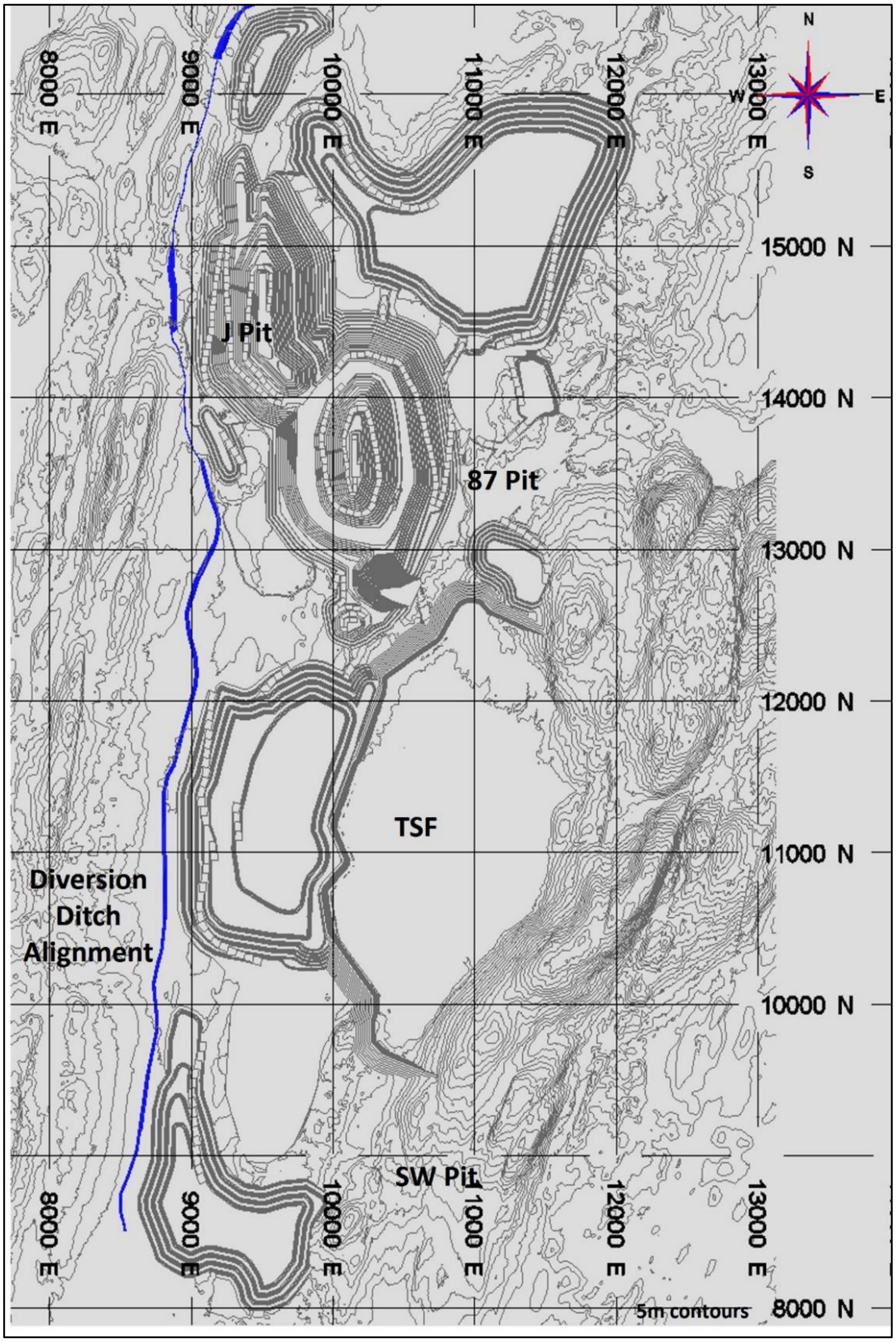
Source: AGP (2024)

Figure 16-24: End of Year 10



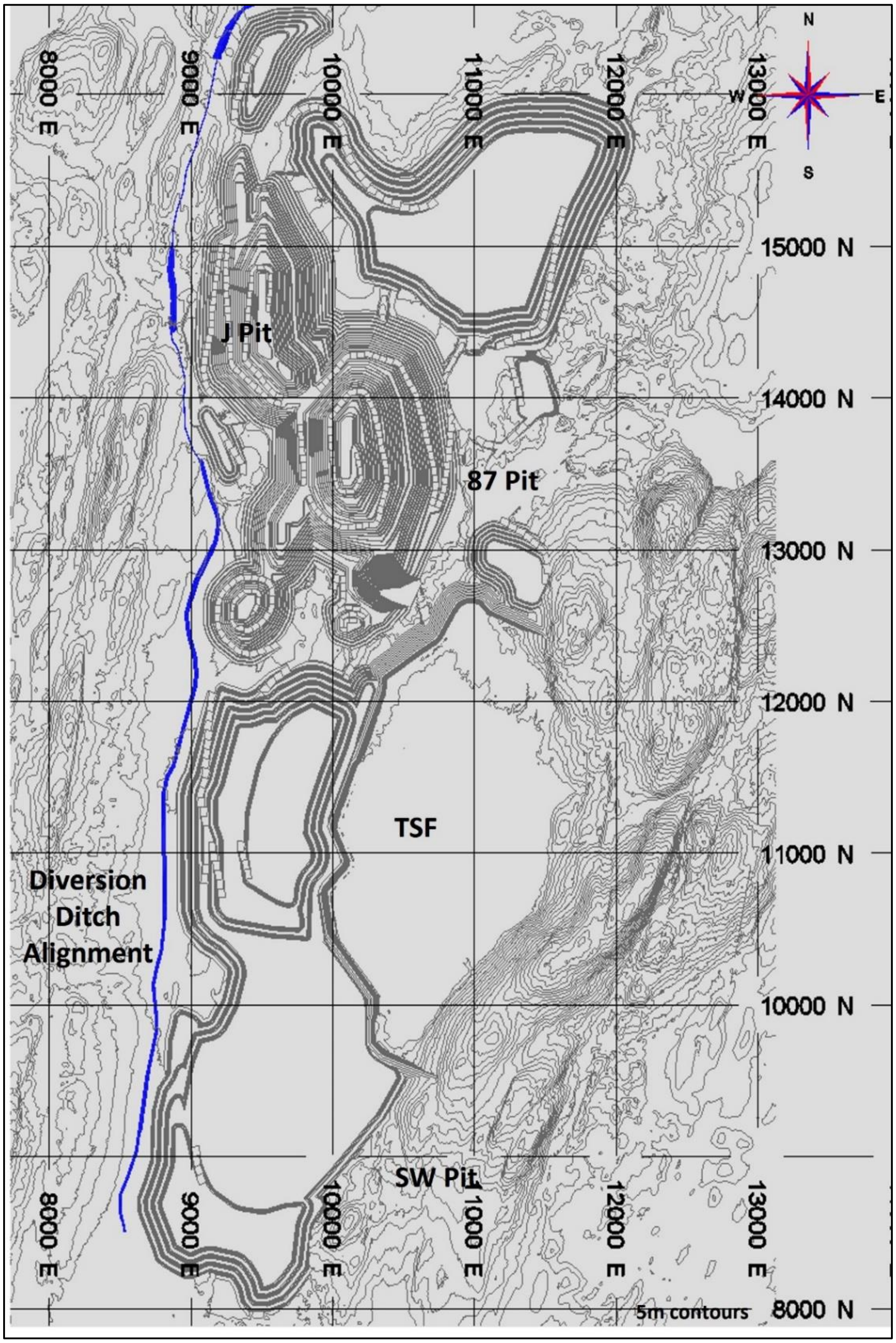
Source: AGP (2024)

Figure 16-25: End of Year 15



Source: AGP (2024)

Figure 16-26: End of Year 21 (Mining Complete)



Source: AGP (2024)

16.3.7 Mine Equipment Selection

The mining equipment selected to meet the required production schedule is conventional mining equipment, with additional support equipment for snow removal and surface ditching maintenance.

Drilling will be completed with down the hole hammer (DTH) drills with a 203 mm bit. This provides the capability to drill 10 metre bench heights in a single pass.

The primary loading units will be 34 m^3 hydraulic shovels. Additional loading will be completed by 23 m^3 loaders. It is expected that one of the loaders will be at the primary crusher for the majority of its operating time. The haulage trucks will be conventional 229 tonne rigid body trucks.

The support equipment fleet will be responsible for the usual road, pit, and dump maintenance requirements. But due to the climatic conditions expected will have a larger role in snow removal and water management. Aggregate production of approximately 500 kt per year will be required for stemming and road crush purposes. Snowplows and additional graders have been included in the fleet. In addition, smaller road maintenance equipment is included to keep drainage ditches open and sedimentation ponds functional.

Additional fleet detail is included in Section 21.

16.3.8 Blasting and Explosives

Blast patterns are the same for feed and waste material. The blast patterns will be 6.1 m x 5.5 m (spacing x burden). Holes will be 10 metres plus an additional 1.1 metres sub-drill for a total 11.1 metres.

The power factor with this pattern size will be 0.31 kg/t. Only emulsion explosives will be used due to the expected wet conditions.

The blasting cost is estimated using quotations from a local vendor. The mine is responsible for guiding the loading process, including placement of boosters/Nonels, and stemming and firing the shot.

Total monthly cost in the service of delivering the explosives to the hole is $383,000/month for the vendor's pickup trucks, pumps, and labour is also applied and covers the cost of the explosives plant. The explosives vendor also leases the explosives and accessories magazines to Troilus as part of that cost. Further explosives details are included in Section 21.

16.3.9 Grade Control

Grade control will be completed with a separate fleet of reverse circulation (RC) drill rigs. They will drill the deposit off on a 10 m x 5 m pattern in areas of known mineralization taking samples each metre. The holes will be inclined at 60 degrees.

In areas of low-grade mineralization or waste the pattern spacing will be 20 m x 10 m with sampling every 5 m. These holes will be used to find undiscovered veinlets or pockets of mineralization.

These grade control holes serve to define the mill feed grade and mineralization contacts.

Samples collected will be sent to the assay laboratory and assayed for use in the short-range mining model. Blasthole sampling will also be part of the grade control program initially to determine the best method for Troilus.

16.3.10 Pit Dewatering

Pit dewatering is an important part of mining at Troilus particularly since the pits will be below the creek level and J and 87 pits are currently full of water. Efficient and cost-effective dewatering will play a role in the Project development. Dewatered slopes may allow a reduction in the strip ratio by permitting steeper inter-ramp angles that would also be inherently safer.

It is estimated that an average groundwater seepage of 4 Mm^3/year and a similar order of magnitude of runoff and will need to be pumped from within the pits (WSP, 2024c); the volumes will vary during the operations. The water will need to be pumped to the environment discharge point near the sedimentation ponds. Storm events have the potential to impact mining operations, and between 0.3 m^3/s and 0.5 m^3/s per pit of pumping capacity are required to manage the in-pit water storage during one of these storm events (WSP, 2024c). The capital cost estimate has considered in Section 21 for the number of pumps required on site to handle such an event.

The dewatering system includes the pumps, sumps, and pipelines responsible for moving water from the pit to the discharge points. Labour for this is already included in the General and Mine Engineering category of the mine operating cost. The mine has a dedicated pump crew and pump crew foreman.

Additional dewatering in the form of horizontal drain holes is also part of the dewatering operating costs. These holes will be drilled in annual campaigns starting in Year -1. The design concept is a series of holes 50 m in length, angled up slightly and drilled into the highwalls. They will allow the water behind the wall to drain freely and prevent pore water pressure build-up particularly during freezing conditions.

16.3.11 Pit Slope Monitoring

Slope movement monitoring will be required during operations. Initial slope monitoring could be conducted with prisms read by manual or automated survey methods. A permanent, automated system will be necessary once operating slope measurements results for the first several years have been gathered and analyzed. At that point, a permanent, fully automated monitoring system using radar should also be considered for ultimate pit walls and some of the interim walls based on the risks identified during operation and prism monitoring results from the first 12 to 24 months of slope performance. If several ultimate pit walls are mined, more than one radar system may be required. Consider two to four radar systems while ultimate pit walls are being mined. This would allow for one or two fixed radars to monitor unanticipated conditions and long-term deformation to complement the prism array as well as one or two mobile radars for short term monitoring of areas of concerns to reduce operational risks with a more optimized positioning of the radar. Detailed slope movement information will be useful for calibrating future numerical models to support detailed pit designs at depth.

A minimum number of vibrating wire piezometers (3 to 4 installations of 200 m to 300 m deep) should be installed around each pit to capture information about the drawdown cones / pore pressure distributions as the pits get deeper. They may also be useful to evaluate effectiveness of installed drains. Horizontal passive drains of 50 m spacing have been included in the costing for the west walls of 87 and J pits. These drains will provide local depressurization to improve slope performance.

Pit wall mapping may be conducted using either digital or physical methods. The mapping results can then be reviewed and interpreted for use in verifying suitability of slope and blast designs.

Operating practices will need to be developed so that blast designs and vibrations are monitored for their impact on pit walls. Equipment operator training is also recommended to ensure scaling and clean-up near walls is completed adequately.

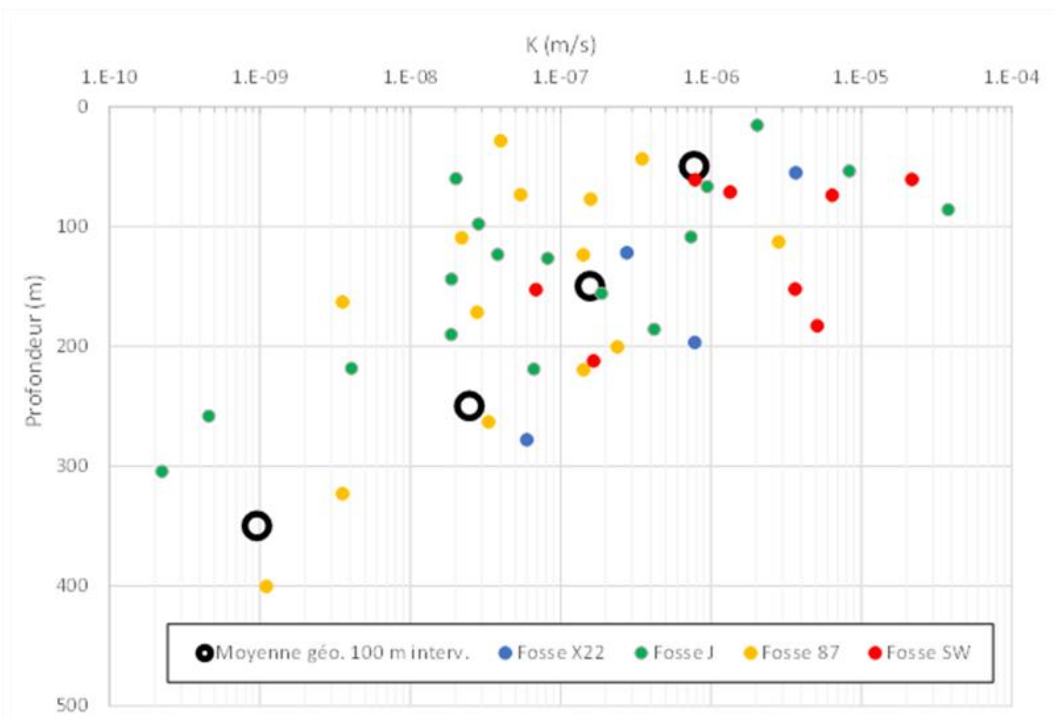
16.3.12 Hydrogeology

A groundwater model was developed to evaluate the discharge of groundwater in the four mine pits (87, J, SW and X22) and the drawdown caused by the dewatering of the pits (WSP, 2024d). The model also includes the derivation of the Bibou Creek diversion channel. The model was developed with the most recent available hydrogeological information at the time of the study execution. The details of the model construction are provided in the following paragraphs.

Hydraulic conductivity

Between 2020 and 2023, two hydrogeological investigation campaigns were undertaken to characterize the hydrogeological properties of the bedrock. Figure 16-27 shows the variation of the hydraulic conductivity with depth for the four pit areas. The black circle indicates the geometric means for each 100 m depth interval. The results show a general decreasing trend of the hydraulic conductivity with depth. A wide range of values is observed in the first 200 m interval.

Figure 16-27: Hydraulic Conductivity vs Depth



Source: WSP, 2024d

Note: The black circles represent the geometric mean per 100 m depth interval.

The geometric mean of the hydraulic conductivity for each of the pit area is shown in Table 16-13. High hydraulic conductivity of the bedrock is observed int the SW pit area and also in X22 pit area.

Table 16-13: Geometric Mean of the Hydraulic Conductivity for the Four Pits Area

Area	K (m/s) – Geometric Mean
Pit J	8×10^{-8}
Pit X22	5×10^{-7}
Pit 87	5×10^{-8}
Pit SW	2×10^{-6}

Groundwater Recharge

The groundwater recharge was estimated by a monthly surface water balance. Recharge flux were estimated for the bedrock outcrop, till, fluvioglacial deposits, tailings, and peat. Recharge fluxes are presented in Table 16-14.

Table 16-14: Estimated Recharge Fluxes Used in the Groundwater Model

Recharge Zone	Average Annual Recharge (mm/y)
Rock outcrop	83
Till	110
Sand/gravel/waste rock	311
Tailings	40
Peat	5

Model Construction

The model bedrock hydraulic conductivity zone is built with four zones of different thicknesses: 6 m to 35 m, 35 m to 150 m, 150 m to 300 m and 300 m to 700 m. The geometric mean of hydraulic conductivity of these intervals was used as the first iteration for the calibration.

The brecciated zones defined in the geological model provided by Troilus in the SW pit area were used as a high-hydraulic conductivity zone. A value of hydraulic conductivity of 1×10^{-6} m/s was used in this zone.

The fracturing caused by the blast on the pit walls during the development of the pit was simulated by adding a ~30 m thick zone behind the pit walls. For this zone, a value of hydraulic conductivity of one order of magnitude higher than the corresponding rock unit was used.

Three simulations were realized. The simulations were done in steady state to represent the long-term average of recharge.

- The first simulation was used to calibrate the model for the current (2022 to 2023) conditions (actual water level in monitoring wells and in pits 87 and J, TSF geometry and waste rock pile geometry). The calibration of groundwater heads was considered satisfactory with a mean error of -0.9 m and a normalized RMSE of 4%. The calibrated hydraulic conductivity for all the geological units is lower than the measured geometric means. Table 16-15 presents the simulated groundwater discharge for pits 87 and J for the actual conditions.

Table 16-15: Groundwater Discharges to Open Pits for the Simulation of the Current (2022 to 2023) Conditions

Pit	Groundwater Discharge (m^3/d)
Pit 87	2130
Pit J	1310

- The second simulation aimed to simulate the final stage of the mine exploitation with the four pits fully excavated and the final waste rock piles geometry. The resulting groundwater discharge in the mine pits is presented in Table 16-16.

Table 16-16: Groundwater Discharges to Open Pits for the Calibrated Model

Pit	Groundwater Discharge (m^3/d)
Pit 87	2450
Pit J	2370
Pit X22	2350
Pit SW	4010

- The third and final simulation aimed to assess the impact of an increase in the hydraulic conductivity of the bedrock to the values of the geometric mean calculate from the measured values (the calibrated values are lower than the geometric means). The simulated groundwater discharge to the four open pits is presented in Table 16-17. This sensitivity simulation should be considered in the upper range of ground water discharge and is presented for comparison only.

Table 16-17: Groundwater Discharges to Open Pits for the Simulation Where Hydraulic Conductivity of the Bedrock is Increased to the Geometric Mean

Pit	Groundwater Discharge (m^3/d)
Pit 87	4360
Pit J	4340
Pit X22	4120
Pit SW	6500

17 RECOVERY METHODS

17.1 Process Design

The Troilus process plant will treat copper-gold ores from four deposits using crushing, grinding and flotation to produce a copper concentrate with gold and silver, with provision for gravity concentration in the future to produce gold doré. The flowsheet is similar to the original Troilus circuit but has been updated to provide a low-cost energy efficient plant, while maximizing gold and copper recoveries.

The key criteria for equipment selection are suitability for duty, safety, reliability, and ease of maintenance. The process plant layout provides ease of access to all equipment for operating and maintenance requirements, whilst maintaining a layout that will facilitate construction progress in multiple areas concurrently.

The key project design criteria for the plant are:

- Nominal throughput of 50,000 t/d (dry).
- Primary and secondary crushing circuit with availability of 75% supported by the use of surge bins and dedicated feeders for choke feeding cone crushers for optimum crushing performance and wear minimization.
- A large covered 12-h live stockpile provides surge capacity between secondary crushing and the remainder of the plant.
- High-pressure grinding rolls (HPGR) as tertiary crushing followed by ball milling at an availability of 88% using standby equipment in critical areas with a reliable grid power supply.
- Sufficient automated plant control to minimize the need for continuous operator interface and allow manual override and control when required.

Study design documents have been prepared incorporating engineering design criteria and key metallurgical design criteria derived from the results of the metallurgical testwork conducted to date.

17.1.1 Selected Process Flowsheet

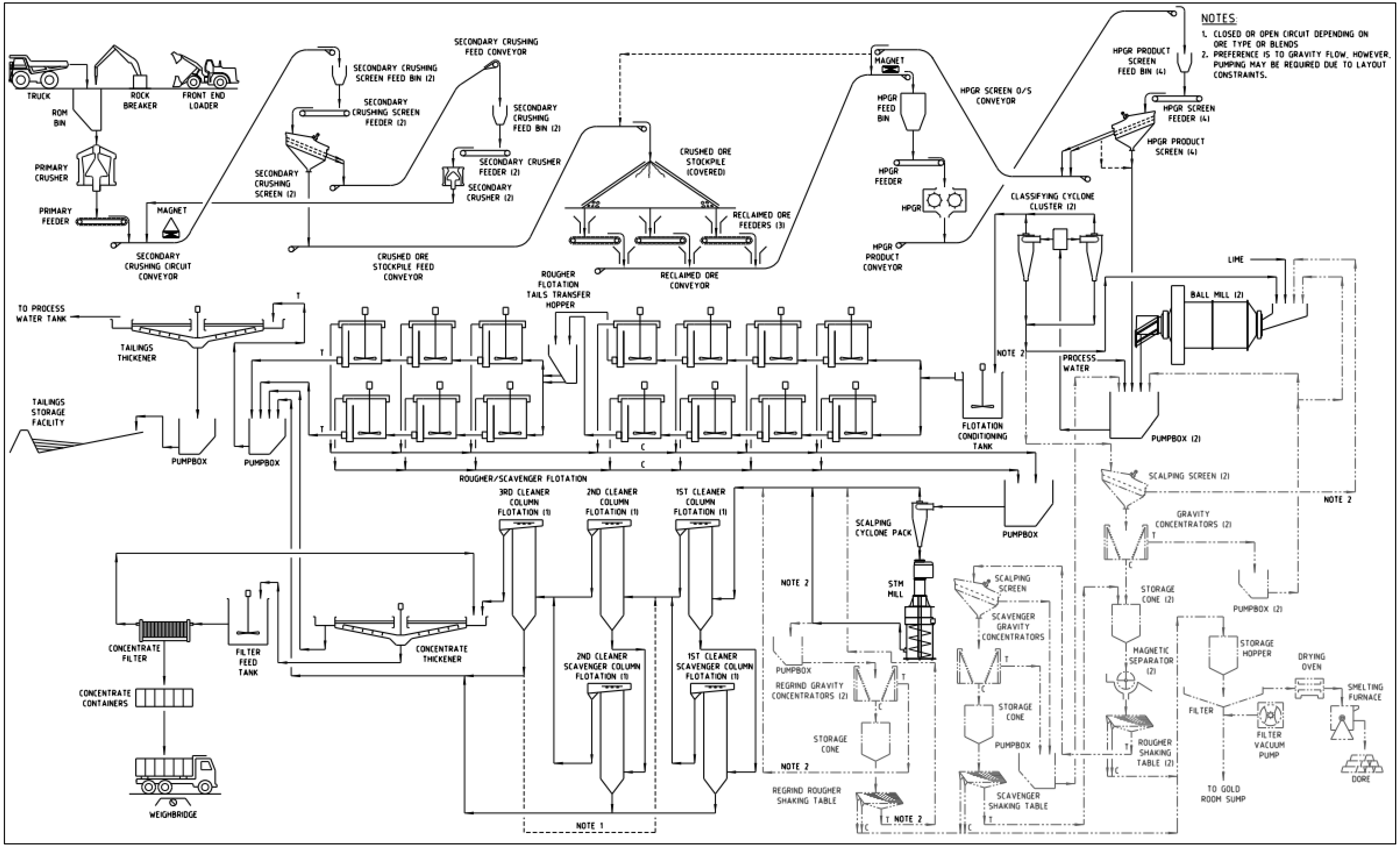
The plant has been designed for a nominal throughput of 50,000 t/d (dry) at design head grade of 0.74 g/t gold, 1.50 g/t silver and 0.09% copper, and life-of-mine (LOM) average head grades of 0.49 g/t gold, 1.00 g/t silver and 0.06% copper. The overall flowsheet includes the following steps:

- two-stage crushing using an open circuit gyratory crusher followed by two parallel closed-circuit secondary cone crushers to produce a -45 mm crushed product for feed to the HPGR; the crushed ore will be stored in a covered stockpile
- a single HPGR operating in closed circuit with four parallel vibrating screens to produce a -5 mm feed to the ball mills
- grinding and classification in two parallel closed ball milling circuits with provision for rougher & scavenger gravity concentration in the future
- bulk rougher and scavenger flotation to produce a primary copper concentrate

- bulk concentrate regrinding in an open-circuit configuration, with provision for regrind gravity concentration in the future
- bulk cleaner flotation, using three stages of column flotation cleaning
- final copper concentrate thickening and filtration
- smelting of rougher scavenger circuit gravity concentrates to produce doré in the future
- tailings thickening of the combined flotation tails and disposal in a tailings storage facility (TSF)

Figure 17-1 presents an overall process flow diagram depicting the major unit operations incorporated in the selected process flowsheet.

Figure 17-1: Overall Process Flow Diagram



Source: Lycopodium 2024

17.1.2 Key Process Design Criteria

The key process design criteria listed in Table 17-1 form the basis of the detailed process design criteria and mechanical equipment list. Design parameters for the process plant are based on testwork conducted in more recent programs (after July 2021) supporting the DFS.

Table 17-1: Key Process Design Criteria

Parameter	Units	Value	Source
Plant Throughput	tpd	50,000	Troilus
Head Grade – LOM / Design	g Au/t	0.49 / 0.74	Troilus
	g Ag/t	1.00 / 1.50	Troilus
	% Cu	0.06 / 0.09	Troilus
Gravity Recovery – Wt. Avg. of Ore Zones	% Au	25.5	Gravity Modelling
	% Ag	3.1	Gravity Modelling
	% Cu	Negligible	Agreed
LOM Overall Recovery – Wt. Avg. of Ore Zones	% Au	92.7	Testwork/Calc.
	% Ag	91.9	Testwork/Calc.
	% Cu	91.8	Testwork/Calc.
Copper Conc. Grade (Range)	% Cu	13-20	Testwork/Calc.
Crushing/Milling/Filtration Plant Utilisation	%	75 / 88 / 85	OMC/Lycopodium
Bond Crusher Work Index (CWi) – est'd from Axb	kWh/t	22.5	Consultant/OMC
Bond Ball Mill Work Index (BW_i) – 85th Percentile	kWh/t	13.8	Consultant/OMC
SMC Axb		26.0	Consultant/OMC
Bond Abrasion Index (A_i)	g	0.290	Consultant/OMC
HPGR Specific Throughput (m-dot)	$t\cdot s/(m^3\cdot h)$	270	Consultant/OMC
HPGR Circuit Feed & Product Size (F_{100}/P_{100})	mm	45 / 5.0	Consultant/OMC
Grinding Product Size (P_{80}) – Primary / Regrind	µm	75 / 20	Testwork/Troilus
Rougher/Scav. Conc. Mass Pull – Test / Design	%	4.1 / 6.0	Testwork/Agreed
Final Flotation Conc. Mass Pull – Test / Design	%	0.36 / 0.41	Testwork/Agreed
Major Reagents - Lime (Total)	kg/t	1.4	Testwork
- SPRI 206 (Rougher/Cleaner)	g/t	28 / 4	Testwork
- PAX (Rougher/Cleaner)	g/t	48 / 8	Testwork
- $N_{a2}SO_3$ (Rougher/Cleaner)	g/t	0 / 400	Testwork
- Frother (Rougher/Cleaner)	g/t	25 / 4	Testwork
Concentrate Thickener Solids Loading	$t/m^2\cdot h$	0.20	Assumed
Tails Thickener Solids Loading	$t/m^2\cdot h$	0.89	Testwork
Tailing Disposal		Pumped to TSF	Troilus

17.2 Crushing Circuit

Primary and secondary crushing circuit will produce -45 mm product to feed the HPGR circuit. The HPGR circuit will produce -5 mm product, suitable for feeding ball mills.

17.2.1 Primary Crushing

The run-of-mine (ROM) ore will either be dumped directly by a dump truck or reclaimed from the ROM stockpile by front-end loader (FEL) into the ROM bin prior to feeding a primary gyratory crusher. A rock breaker will be provided to assist in breaking down oversize rocks as required. Crushed product from the gyratory crusher will be drawn from the crushed ore bin beneath the crusher by an apron feeder and discharged onto the secondary crushing circuit conveyor.

This conveyor will be fitted with a weightometer to monitor the plant feed rate and control the secondary crushing fresh feed rate via the apron feeder variable speed control. A tramp metal magnet at the discharge of the conveyor will remove metal from the crushed ore stream via a tramp metal transfer chute to a scrap metal bunker.

The primary crushing area will be serviced by the primary dust collection system comprising of a series of extraction hoods, ducting, bag house, and dust collection bin.

17.2.2 Secondary Crushing and Screening

Primary crushed ore will report to a secondary crushing circuit consisting of a screen feed bin and two parallel belt feeders to withdraw the material to feed onto two parallel vibrating screens. These double deck screens will be fitted with 75 mm (top) and 45 mm (bottom) aperture screening panels. Screen oversize will be conveyed to the secondary crushing feed bin where material is drawn onto two belt feeders to feed two cone crushers. Secondary crusher product will be conveyed back to the secondary crushing circuit conveyor. The -45 mm screen undersize will be conveyed to a covered stockpile. The stockpile provides a break and surge capacity between the secondary crushing and the remainder of the downstream process.

The secondary crushing feed conveyor (screen oversize) will be equipped with a self-cleaning belt magnet which will discharge the tramp metal to a scrap metal bunker. A metal detector will also be installed after the magnet to identify non-magnetic metal. On activation of the metal detector, the belt will automatically be stopped, and the metal location will be flagged via a paint marker for manual removal by an operator.

The secondary crushing area will be serviced by two dust collection systems, each comprising of a series of extraction hoods, ducting, bag house, and dust collection bin.

17.2.3 Stockpile Reclaim and HPGR

Secondary crushed ore will be stored at a covered stockpile and reclaimed via three apron feeders in the tunnel beneath to discharge onto a reclaimed ore conveyor. The conveyor will be equipped with a weightometer to monitor and control the total feed rate to the HPGR circuit.

The reclaimed crushed ore will report to a surge bin and a feeder ahead of a single HPGR in closed circuit with four wet screens. HPGR product will be transferred to the HPGR screen feed bin which will distribute to four parallel double-deck multi-slope screens via four belt feeders. The screens will each be equipped with a top deck aperture of 8 mm and a bottom deck aperture of 5 mm. The screen middlings and oversize will be returned to the reclaimed ore conveyor through the stockpile diverter gate, whereas the screen undersize will report as the HPGR product to the cyclone feed hoppers in the grinding circuit. The expected product size from the HPGR circuit will be P_{80} 2.7 mm.

A circulating load of 85% was selected for the HPGR circuit based on testwork and OMC's experience.

The HPGR area will be serviced by a dust collection system, comprising of a series of extraction hoods, ducting, bag house, and dust collection bin. A vertical spindle sump pump at the bottom of the building will remove all spillage and flooding from this area.

The HPGR screens oversize conveyor will be fitted with a weightometer to monitor the total screen oversize tonnage to allow the HPGR fresh feed and feed to the grinding circuit to be calculated (i.e., total tonnage to HPGR minus HPGR screen oversize).

17.3 Grinding Circuit

HPGR circuit product at -5 mm will flow via launders to two cyclone feed hoppers feeding two parallel ball milling circuits.

Each milling circuit will feature a single-stage ball mill operating in a closed circuit with a cyclone cluster. Each ball mill will be an 8.23 m diameter x 13.34 m effective grinding length (EGL). The overflow mills will be equipped with trommels and driven by dual pinions, each connected to a 9 MW variable speed drive motor. Water will be added to the feed chute to achieve the desired milling density. Lime will also be added directly into the mill feed spout from a lime silo via a dedicated screw feeder to each mill.

Product from the ball mill will discharge over a trommel with oversize reporting to a scat bunker, where it will be periodically removed returned to the mill feed via an FEL. Trommel undersize will flow by gravity to the cyclone feed hopper where it will be further diluted to achieve the required cyclone feed density. Cyclone underflow will return to the ball mill, while cyclone overflow at a P_{80} of 75 µm will flow by gravity to the conditioning tank ahead of the rougher flotation circuit. Provision has been made for a portion of the cyclone underflow stream from each milling train to be diverted to a gravity concentration circuit in the future.

A vertical spindle sump pump will service the floor area of each ball milling circuit. The concrete floor under each mill area will slope to the sumps to facilitate clean-up.

Makeup grinding media for the ball mills will be added using a dedicated ball charging kibble lifted by the grinding area maintenance crane.

17.4 Gravity Concentration Circuits (Future)

In the future, there will be two gravity circuits – one in the ball mill circuit and one in the concentrate regrind circuit. For the ball mill circuit, each grinding line will divert part of the underflow stream from the classifying cyclones to the gravity concentration circuit. The aim of the gravity circuit is to recover free coarse gold and gold associated with coarse sulphide particles.

Process water will be added into the cyclone underflow launders to facilitate flow to the gravity scalping screens and to maintain a suitable %solid (w/w) to the gravity concentrators. The vibrating gravity scalping screens will remove coarse particles and trash from the slurry stream ahead of the gravity concentrator. The screen oversize will be recycled to the ball mill while the screen undersize will pass through to feed the centrifugal gravity concentrators.

Each of gravity concentrator will operate in a semi-continuous mode where the cycle consists of 45 minutes continuous concentration followed by periodic flushing of the accumulated concentrate to a storage cone for further concentration with shaking tables. Process water will be used to fluidize the feed to the gravity concentrator. The gravity tailings will return to the cyclone feed pump box in the grinding circuit.

Concentrate from the gravity concentrate storage cone in each train will be pumped to a magnetic separator to remove metal pieces prior to being processed through the rougher shaking table. The tails from the rougher shaking table in both trains will be fed to a scavenger shaking circuit, consisting of another scalping screen, a centrifugal gravity concentrator and another shaking table. The tails from the scavenger shaking table circuit will return to the rougher shaking table feed while the concentrate will be combined with the rougher shaking table concentrate in both trains, as well as the regrind shaking table concentrate into a storage hopper to be transferred to the gold room for filtering, drying, and smelting into gold doré bars.

In the concentrate regrind circuit, a pump box will be installed in the future to collect the slurry discharging from the regrind mill to pump to two gravity concentrators. Concentrate from the gravity concentrators will be further concentrated by a shaking table. The regrind shaking table concentrate will be transferred with other concentrates from the rougher and scavenger shaking tables to the gold room for filtering, drying, and smelting into gold doré bars. The tails from the regrind shaking table will be bled out of the regrind circuit by diverting it to the cleaner flotation circuit conditioning tank.

17.5 Rougher and Scavenger Flotation Circuit

Cyclone overflow from both trains of milling circuit will report to a single common rougher/scavenger flotation conditioning tank with approximately 6 minutes of residence time. Frother and collector will be metered into the conditioning tank. Lime will be needed to adjust the pH, with the majority being added to the mill feed, as detailed in Section 17.3. Process water will also be added if required to dilute the feed to the appropriate slurry density. The resultant slurry will report to two trains of rougher/scavenger flotation cells.

The rougher/scavenger flotation section will consist of two trains of seven Stack Cell units in series. Rougher and scavenger concentrate will flow by gravity to the regrind circuit to further reduce the grind size to P_{80} 20 µm to feed the cleaning flotation circuit.

The rougher/scavenger tailings will be combined with tails from the cleaning circuit to the tailings thickener feed box.

Pressure pipe samplers will be installed on the various points to take samples from two rougher/scavenger tailings lines and from the combined rougher/scavenger concentrate line to the on-stream analyser (OSA) for process control purposes.

The flotation building gantry crane will be used for all maintenance lifting functions within the flotation area. Sumps with vertical spindle sump pumps will service this area for spillage clean-up.

17.6 Regrind Circuit

Rougher/scavenger concentrate will be transferred to an opened regrind circuit to reduce the grind size to P_{80} 20 µm prior to feeding the cleaner flotation circuit.

Concentrate slurry in the regrind cyclone feed pump box will be pumped to a cyclone cluster with the cyclone underflow feeding a vertical stirred regrind mill. The cyclone overflow at P_{80} 20 µm joined by the regrind mill discharge will flow by gravity to the cleaner conditioning tank.

Pressure pipe samplers will be installed on the cyclone feed and on the feed line to cleaner conditioning tank to send samples to the on-stream analyser (OSA) for process control purposes.

The regrind media will be introduced via the bulk regrind media hopper. The bulk media hoist will be installed to allow filling of the bulk regrind media hopper from bulk bags.

A sump with vertical spindle sump pump will service this area for spillage clean-up.

17.7 Cleaner Flotation Circuit

Cleaner flotation will consist of three stages of open circuit cleaning with the option to recirculate the cleaner 3 tails back to cleaner 2 as required for certain ore types. Column cells will be used in all three cleaner stages to reduce gangue entrainment into the concentrate.

Two conditioning tanks will be installed in series a head of each cleaning stage for sequencing of the reagent dosing. The pH of the slurry should have already been adjusted ahead of the cleaner flotation circuit. Milk-of-lime can also be added to the feed slurry before each cleaning stage if required. In the first conditioning tank of each cleaner stage, sodium sulphite (Na_2SO_3) will be added to depress pyrite. Zinc sulphate ($ZnSO_4$) and sodium trithiocarbonate (Na_2CS_3) may also be added to depress zinc and arsenic, respectively, if required. In the second conditioning tank of each cleaner stage, collectors such as PAX and SPRI 206 will be added. Frother will also be added to each cleaner column recirculation pump suction if required. The capability to add process water to dilute the slurry to the desired feed density will also be provided.

The first and second stages of cleaning will each consist of a cleaner and a cleaner scavenger column. The third cleaner stage will consist of only a cleaner column.

Cleaner 1 tails will be pumped to feed the cleaner 1 scavenger column via a transfer hopper and pump. Cleaner 1 and cleaner 1 scavenger concentrates will be pumped to the cleaner 2 circuit by a froth pump. Cleaner 1 scavenger tails will be transferred to the flotation tails pump box also via a transfer hopper and pump. Pressure pipe samplers will be installed on both cleaner 1 concentrate and tails line to send samples to the OSA for process control purposes.

Cleaner 2 tails will be pumped to feed the cleaner 2 scavenger column via a transfer hopper and pump. Cleaner 2 and cleaner 2 scavenger concentrates will be pumped to the cleaner 3 circuit by a froth pump. Cleaner 2 scavenger tails will be transferred to the flotation tails pump box also via a transfer hopper and pump. Pressure pipe samplers will be installed on both cleaner 2 concentrate and tails line to send samples to the OSA for process control purposes.

Cleaner 3 tails will be transferred to the flotation tails pump box via a transfer hopper and pump. Cleaner 3 concentrate will be pumped to the concentrate thickener via a froth pump. Pressure pipe samplers will be installed on both cleaner 3 concentrate and tails line to send samples to the OSA for process control purposes.

Sumps with vertical spindle sump pumps will service this area for any spillage clean-up.

17.8 Gold Room (Future)

Gold and silver bearing gravity concentrates from the future rougher, scavenger, and regrind shaking table circuit will be collected in the shaking table concentrate storage hopper, where it will be filtered using a vacuum sludge pan filter and dried in an oven.

Once the dried concentrate cake has cooled, it will be mixed with fluxes and loaded into the smelting furnace. The fluxes will react with base metal oxides to form a slag, whilst the gold and silver will remain as molten metal. The molten metal will be poured into moulds, to form doré ingots, which will be cleaned, assayed, stamped, and stored in a secure vault ready for dispatch. The slag produced will periodically be returned to the ball mill feed chute by hand.

The smelting furnace will be required to process the expected mass of concentrate for the design head grade case at a frequency of 1 smelt per day (7 smelts/week) and with the ability to process up to 2 smelts a day (14 smelts per week) if needed.

Fume extraction/vents and a furnace bag filter will be provided to remove noxious gases / dust from the smelting furnace. In addition to this, fresh air fans will be provided to ensure there is adequate ventilation inside the gold room.

A sump with a vertical spindle type sump pump, coupled with a gold trap, will be installed in the gold room to remove any hose down or spillage.

Auxiliary equipment for the gold room will include:

- flux bin, platform scale, flux mixing table
- gold pouring cascade trolley and slag cart
- bullion moulds and bullion cleaning table
- bullion balance

17.9 Concentrate Thickening and Filtration Circuit

Final (cleaner 3) concentrate will be pumped to a 9 m diameter high rate concentrate thickener. Flocculant stock solution will be further diluted with bulk circuit water (recirculated from thickener overflow) in an in-line static mixer prior to dosing into the concentrate thickener. Thickener overflow will be collected in an overflow tank and pumped to the tailings pump box for disposal.

Concentrate thickener underflow, at approximately 55% solids w/w, will be pumped to the agitated concentrate filter feed tank by the concentrate thickener underflow pump. This tank will provide 48 h of surge capacity between the concentrate thickener and the bulk concentrate filter press.

A sump and vertical spindle sump pump will be provided to return spillage to the bulk concentrate thickener.

Thickened concentrate will be pumped to the concentrate filter press. The filter press will remove water from the concentrate to meet the target moisture of approximately 9% w/w using a series of pressing and air blowing steps. After the desired filtration time, the filter press will open and discharge bulk concentrate directly to a concentrate bunker. Following discharge of concentrate, the filter cloth will be washed prior to the next cycle using process water. Filtrate from the bulk concentrate filter will be returned by gravity to the concentrate thickener.

Filter cake will be loaded by a front-end loader (FEL) into 20-ft. shipping containers and weighed before being shipped off-site by a truck. Samples will be taken from each batch for process control and metallurgical accounting purposes.

A sump and vertical spindle sump pump will be provided to return any spillage from the bulk filter area to the bulk concentrate thickener.

17.10 Tailings Disposal

Flotation tailings along with sumpages from the flotation, flocculant, frother and collector (SPRI 206) areas will be transferred to the tailings thickener feed box ahead of a 64 m diameter high-rate thickener to dewater the slurry prior to pumping to the tailings storage facility (TSF).

Tailings thickener underflow, at a target density of 55% solids (w/w), will be transferred to the tailings pump box and pumped to the TSF by the tailings pump. Tailings thickener overflow will flow by gravity to the process water tank for reused within the process plant.

A dedicated sump and vertical spindle sump pump will be provided to return spillage to the tails thickener.

17.11 Reagents and Consumables

Reagents are stored in bulk on site at a central location for 7 days of inventory.

17.11.1 Flocculant

Flocculant will be delivered to site in 1 t bulk bags and stored in the reagent shed. A vendor supplied flocculant mixing system will be installed, which will include the flocculant hopper, flocculant bag breaker, flocculant blower, wetting head, flocculant mixing tank and transfer pump. Powdered flocculant will be loaded into the flocculant hopper by hand and pneumatically transferred into the wetting head, where it will be contacted with raw water. Flocculant solution will be agitated in the flocculant mixing tank for a pre-set period. Afterwards, the flocculant solution will be transferred to the flocculant storage tank.

Flocculant will be dosed to the concentrate and tailings thickeners using variable speed positive displacement pumps. Flocculant will be further diluted prior to the addition point at the thickeners.

A dedicated sump and vertical spindle sump pump will be provided in this area.

17.11.2 Lime

Quicklime (lime) will be added to the process through both a primary and a secondary system. The primary system will add most of the lime required for raising the slurry pH directly into the ball mills while the secondary system will consist of a lime slaker and ring main distribution system which will be used for fine tuning the pH in the flotation circuit.

Primary System:

Dry powder will be delivered by a bulk lime tanker and stored in a silo from where it will be conveyed pneumatically to a small hopper above the mill feed spout for each milling train. A rotary valve will dispense lime at a controlled rate into the ball mill feed spout.

Secondary System:

The secondary lime system will be a mix tank/batch type of lime slaker system. Lime in 1 t bulk bags will be hoisted to the lime silo for storage. Lime will be extracted from the silo via a rotary valve and screw feeder to be mixed with raw water in a lime mixing tank to achieve the desired lime density. Mixed or slaked lime will be stored in the milk-of-lime storage tank to be distributed to the flotation area by the lime slurry circulation pump and a ring main, with take-offs distributing lime as required.

A dedicated vertical spindle sump pump will be provided for spillage control in the slaker area.

17.11.3 Frother

Frother will be delivered in intermediate bulk containers (IBCs) and stored in the reagent shed until required. A permanent bulk box will be installed to provide storage capacity local to the flotation area. Frother will be dosed as received, without dilution. Multiple diaphragm dosing pumps will deliver frother to the required locations within the flotation circuit. Top up of the permanent bulk box will be carried out manually as required.

A single air operated diaphragm sump pump will be provided for spillage control in the frother area.

17.11.4 Collector 1

Potassium Amyl Xanthate (PAX) will be delivered in granular powder form in 850 kg bulk bags and stored in the reagent shed until required. Bags will be lifted into a bag breaker located at the top of the tank with an overhead crane common for the reagents area. The powder will drop into the tank and be dissolved in raw water to achieve the required dosing concentration.

PAX solution will be transferred to the PAX collector storage tank using a centrifugal pump. PAX solution will be delivered to the flotation circuit using diaphragm dosing pumps.

Both the mixing and storage tanks and the sump will be ventilated by duty and standby exhaust fans to remove carbon disulphide gas. A dedicated sump with vertical spindle sump pump will be provided for spillage control.

17.11.5 Collector 2

Collector 2 or SPRI 206 will be delivered in IBCs and stored in the reagent shed until required. A permanent bulk box will be installed to provide storage capacity local to the flotation area. SPRI 206 will be dosed as received, without dilution. Two diaphragm dosing pumps will deliver the reagent to the required locations within the flotation circuit. Top up of the permanent bulk box will be carried out manually as required. A sump with a single air operated diaphragm sump pump will be provided for spillage control in the SPRI 206 area.

17.11.6 Pyrite Depressant

Sodium sulphite (Na_2SO_3) will be delivered in white crystal powder in 1.25 t bags and stored in the reagent shed until required. Bags will be lifted into a bag breaker located at the top of the tank with an overhead crane common for the reagents area. The powder will drop into the mixing tank and be dissolved in raw water to achieve the required dosing.

Na_2SO_3 solution will be transferred to a storage tank using a centrifugal pump. Na_2SO_3 solution will be delivered to the flotation circuit using a diaphragm pump.

Both the mixing and storage tanks, and sump will be ventilated using duty and standby fans to remove carbon disulphide gas. A dedicated sump with a vertical spindle sump pump will be provided for spillage control.

17.11.7 Zinc Depressant

Zinc sulphate ($ZnSO_4$) will be delivered in the form of zinc sulphate monohydrate ($ZnSO_4 \cdot H_2O$) white crystal powder in 1 t bulk bags and stored in the reagent shed for use as required. Bags will be lifted into a bag breaker located at the top of the tank with an overhead crane common for the reagents area. The powder will drop into the tank and be dissolved in raw water to achieve the required dosing concentration.

$ZnSO_4$ solution will be transferred to the storage tank using a centrifugal pump. $ZnSO_4$ solution will be delivered to the flotation circuit using a diaphragm pump. Zinc depressant is typically not required, and this system is only installed as a back-up in the case that certain ore requires zinc depressant.

Both the mixing and storage tanks, and sump will be ventilated using duty and standby fans to remove carbon disulphide gas. A dedicated sump with a vertical spindle sump pump will be provided for spillage control.

17.11.8 Arsenic Depressant

Sodium trithiocarbonate (Na_2CS_3) will be delivered in the form of technical grade powder in 1.2 t bulk bags and stored in the reagent shed for use as required. Bags will be lifted into a bag breaker located at the top of the tank with an overhead crane common for the reagents area. The powder will drop into the tank and be dissolved in raw water to achieve the required dosing concentration.

Na_2CS_3 solution will be transferred to the storage tank using a centrifugal pump. Na_2CS_3 solution will be delivered to the flotation circuit using a diaphragm pump. Arsenic depressant is typically not required, and this system is only installed as a back-up in the case that certain ore requires arsenic depressant.

Both the mixing and storage tanks, and sump will be ventilated using duty and standby fans to remove carbon disulphide gas. A dedicated sump with vertical spindle sump pump will be provided for spillage control.

17.12 Water Distributions

This plant requires process water, raw water, potable water, fire water, and TSF decanted water.

17.12.1 Process Water

Process water used in the process plant will be sourced from different locations including TSF decant water, tails thickener overflow, and from the raw water pumps as make-up if the other two sources are depleted. The process water tank will have a storage capacity of 3,413 m^3 to provide 0.5 h of surge capacity based on average consumption of 6,825 m^3/h.

Process water will be used for the following duties:

- wet screen pulping
- milling circuit sprays and dilution
- gravity concentrator fluidization (future)
- flotation circuit dilution water
- flocculant dilution water
- concentrate filter wash water
- gold room wash water (future)
- plant washdown

17.12.2 Raw Water

Raw water supplied from Lake A will be delivered to a 1,144 m^3 raw water tank for storage. This tank will also be used as a source of fire water with 864 m^3 dedicated as fire water reserve and the remainder to provide 1 h of surge capacity for raw water. Approximately 280 m^3/h of raw water is required on average with a peak intermittent consumption of around 514 m^3/h.

Raw water will be used for the following duties:

- ball mill seal water
- HPGR screen sprays
- reagent mixing
- gold room wash down (future)
- filtered raw water as gland water
- process water make-up via the raw water pump at the raw water source (Lake A)

17.12.3 Potable Water

The potable water treatment plant will be a vendor package. The plant potable water tank will be used to store potable water for use in the OSA, operation facilities and process plant safety showers. A separate safety shower water storage tank and ring main system will be installed to provide water to the safety showers and drinking fountains around the plant.

17.12.4 Fire Water

Fire water will be delivered using a vendor package which will include an electric fire water pump, a fire water jockey pump, and a diesel fire water pump.

17.12.5 TSF Decanted Water

Tailings will be pumped to the TSF. Supernatant solution will be reclaimed from the TSF and recycled to the plant's process water tank by decant return water pumps.

17.13 Other Services and Utilities

17.13.1 Particle Size Analyser

Particle size on the cyclone overflow stream from each milling train will be monitored by a particle size analyser (PSA) to ensure that the targeted grind size has been achieved for the feed to the flotation circuit. The reject from the particle analyser will be sent to the on-stream analyser to monitor the feed stream to rougher flotation.

17.13.2 On-Stream Analyser

The performance of the flotation circuit will be monitored by a dedicated on-stream analyser (OSA) system to allow the operator to adjust air, slurry level or reagent dosages based on real time assays of all the major streams. Analysis will include percent solids, pH, copper, gold, and silver assays. Gold measurement will be inferred based on other elements. Cumulative shift samples for laboratory analysis will also be collected via the OSA sampling system. The system will have a stand-alone control, calibration and reporting system but will have the capacity to provide assay data to the plant control system if required.

Samples will be collected using a combination of sample pumps, pressure pipe samplers and linear samplers as required. Samples will be logically combined after analysis and returned to an appropriate location using vertical spindle style pumps.

17.13.3 Samplers

Different streams are sent to the PSA and OSA systems using various samplers for either metallurgical accounting or process control purposes.

17.13.4 High- and Low-Pressure Air

High-pressure air at 750 kPa will be provided by two high-pressure (rotary) air compressors, operating in a lead-lag configuration. The entire high-pressure air supply will be dried and can be used to satisfy both plant air and instrument air demands, however, the instrument air has additional filters and dryers as an added protection against moisture for the instruments. Dried plant air will be distributed via the main plant air receiver, with additional receivers in the crushing, grinding, flotation, and the future gold room areas.

Low-pressure air at ~230 kPa for the rougher/scavenger flotation circuit will be supplied by two centrifugal type duty and standby air blowers.

Compressed air at 550 kPa for the flotation columns will be supplied from the flotation air receiver with an air pressure regulator to step down from 750 kPa to 550 kPa for the columns.

17.14 Energy Requirements

The estimated annual power consumption is summarized in Table 17-2.

Table 17-2: Average Annual Power Usage

Area	Annual Power Consumption (MWh)
Area 120 - Feed Preparation / Crushing	37.1
Area 130 - Reclaim, Milling & Classification	338.8
Area 130 - Reclaim, Milling & Classification (future with gravity circuit included)	345.4
Area 150 - Flotation & Concentrate Handling	52.5
Area 170 - Gold Room & Smelting (future with gravity circuit included)	0.98
Area 180 - Tails Handling	8.5
Area 210 – Reagents	1.0
Area 220 - Water Services	11.9
Area 230 - Plant Services	0.49
Area 240 - Air Services	8.2
Area 250 - Fuel Storage & Distribution	0.37
Area 310 - Infrastructure-General	0.016
Area 330 - Raw Water Supply	0.97
Area 350 - Tailings Decant System	3.5
Area 360 - Buildings - Admin and Security	2.9
Area 370 - Buildings - Plant Offices	9.3
Area 380 - Permanent Accommodation	136.8
Area 450 - Mining Facilities	0.28
Total	**612.9**
Total (Future with Gravity Circuit Included)	**620.4**

18 PROJECT INFRASTRUCTURE

18.1 General

The Project will comprise of the following infrastructure and on-site facilities that will be newly constructed. These facilities are displayed in Figure 18-1.

- southwest open pit
- waste management facilities (WMF)
- tailings management facility (TMF)
- tailings and reclaim pipelines
- Lake A raw water pipeline
- mine haul roads
- site access roads
- fuel management facility
- primary and secondary crushing
- coarse ore stockpile and reclaim
- process facilities - crushing, milling, thickening, flotation, gravity gold recovery (future), gold room (future), concentrate handling and loadout, water management systems, reagents systems and reagents storage
- site buildings - site administration building, site washrooms, high security building, first aid clinic, gate house, site workshop and warehouse, control room, metallurgical and assay laboratory
- mine services buildings - mine truck shop, mine warehouse, tire change facility, truck wash facility, and mine washrooms
- camp facilities
- explosives storage facility
- sanitary landfill
- water treatment plant (WTP)
- sewage treatment plant (STP)

In addition to the newly constructed facilities, there is existing infrastructure that will be repurposed for the project. Table 18-1 provides a list of the existing infrastructure with the proposed repurposing.

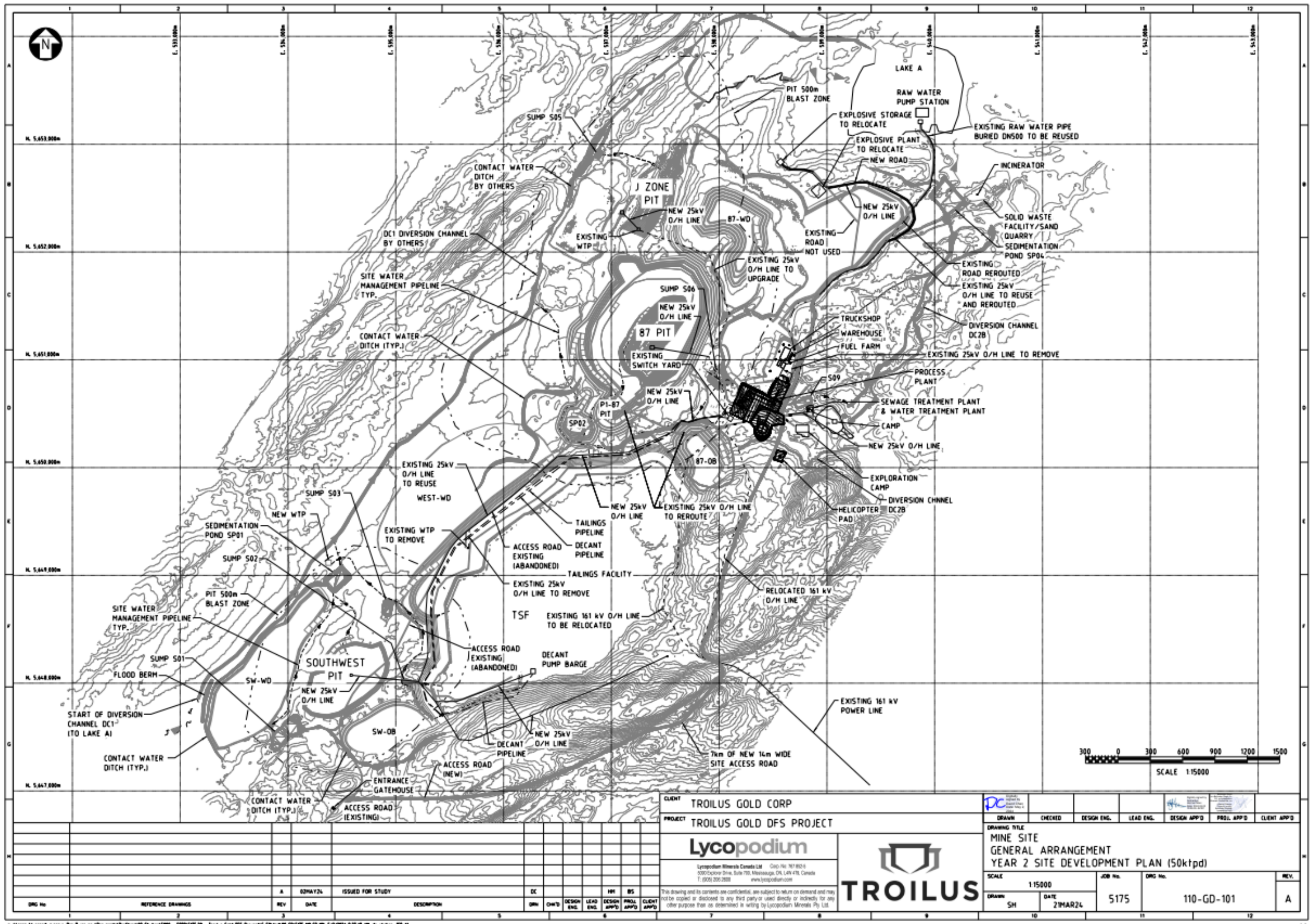
- J-zone open pit
- 87 open pit
- site access roads
- exploration camp
- main substation

- main and in-plant overhead power lines
- potable water wells, pump, and pipeline
- laydown areas
- water management ponds
- tailings storage facility (TSF)
- TSF Water Treatment Plant
- helicopter pad

Table 18-1: Existing Infrastructure

Location	Existing Infrastructure	Re-Purposing
J4 Open Pit	Pumps and e-house	Re-locate and use pumps and e-house for Lake A freshwater
HV Substation	Existing 50 MVA	Expand to 75 MVA
Incoming Powerline	Existing 161 kV	Re-use and re-align along the east side of TSF
Site Powerlines	Existing 25 kV	Re-use and expand to SW pit, remote pump stations and camp.
Roads	Site access roads	Re-use and extend Seven km to the new process plant location
Roads	Site roads	Re-use all existing site roads; re-align and extend to new process plant location
TSF	Existing	Future raises required
TSF Water Treatment Plant	Existing	Adequate for the first two years of operation
Helicopter Pad	Existing	Re-use

Figure 18-1: Overall Site Plan



Source: Lycopodium (2024)

18.2 Roads and Earthworks

18.2.1 Site Access Road

The Project is located 170 km north of Chibougamau. From Chibougamau, the Project is accessed by driving on Highway 167, Route du Nord, and the site access road. Highway 167 is paved and in good condition. The Route du Nord and site access road are gravel and are well maintained year-round. There is a weight limit of 130 t at the Troilus River bridge. The site access road will require a 7 km long realignment around the east side of the TSF to accommodate the new SW open pit. This road will be 14 metres wide and connect the existing road north-east of the process plant that leads to Lake A.

18.2.2 Site Roads

There are several existing internal roads which will provide access between the crushing plant, process plant, mine services, LAKE A and the other site facilities including the administrative buildings and camp. Minor adjustments will be made to the existing internal roads to link new areas of the site. The internal roads will generally be 8 m wide and will be constructed using mine waste. Ditches and culverts will be used where necessary to ensure safe driving conditions.

A network of mine haul roads will be constructed and maintained by the mining department and used to access the open pit, WRFs and for the transport of ore to ROM pad.

18.2.3 Earthworks

The earthworks design for the site development provides an optimized layout and encompasses:

- definition of access and platform limits and elevations
- cut and fill volume calculations
- plan and profile drawings

Earthworks designed for access to buildings are considered the minimum width required for vehicles using the area.

The cut and fill slope parameters adopted for this Project are based on a review of the preliminary geotechnical reports.

- cut batter slopes: 1.0 H : 2.0 V
- fill batter slopes: 1.0 H : 3.0 V

For cuts greater than 8 m deep, a 4 m wide bench will be provided at 8 m intervals.

In general, all areas of the Project have been graded to provide adequate surface drainage collection and management thereby minimizing adverse environmental impact. Slopes will be re-vegetated to control erosion.

18.2.4 Drainage

The site drainage system design is based on rainfall data, the individual drainage areas, an appropriate run-off coefficient to determine the volumes of water to be managed and to size the ditches and culverts.

18.3 Primary and Secondary Crushing, Crushed Ore Stockpile

The primary and secondary crushing facility is located at the northeast end of the process plant area. This facility is shown in Figure 18-2.

The facilities comprise the following major equipment as listed in Table 18-2.

Table 18-2: Primary, Secondary Crushing and Crushed Ore Stockpile Major Equipment

Equipment	Size	Power (kW)
Run-of-mine (ROM) bin	360 t live capacity or 1.5 mine trucks	N/A
Primary crusher	2,778 dry t/hr, MKII 60-89	745
Primary feeder	3,333 dry t/hr (2,100 m wide X 6.9 m long)	90
Secondary crushing circuit conveyor	7,468 dry t/hr (2,400 mm W X 276 m long x 39 m lift)	2 x 560
Secondary crushing screen feed bin	486 m^3 live capacity	N/A
Secondary crushing screens	2 x Double deck multi slope (4.2 m x 8.5 m)	110
Secondary crushing feed conveyor	3,653 dry t/hr (1,800 mm W X 195 m long x 18 m lift)	400
Secondary crushing feed bin	288 m^3 live capacity	N/A
Secondary crushers	1,725 dry t/hr, 2 x MP 800 cone crushers	933 each
Coarse ore stockpile feed conveyor	4,051 dry t/hr (1800 mm W X 260 m long X 44 m lift)	800
Coarse ore stockpile	13.6 hrs live capacity (35 m high X 90 m diameter) 28,400 tonne live	N/A

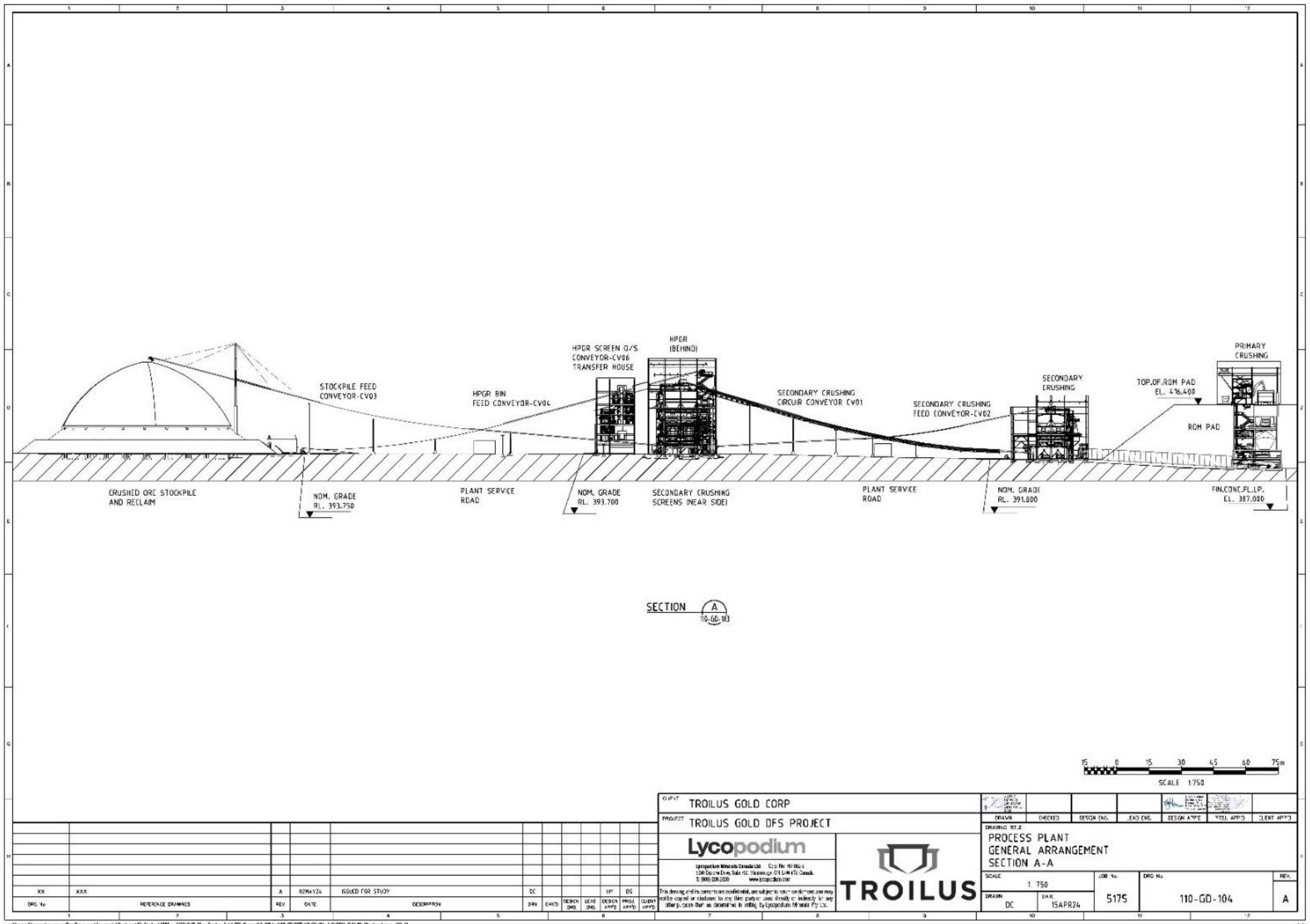
The primary crusher facilities will be contained in a reinforced concrete structure cut into the hill northeast of the process plant. The dimensions of the primary crusher structure are approximately 26 m X 12 m wide X 29 m high.

Mine haul trucks will dump the ore directly into the ROM bin which feeds the primary gyratory crusher. Crushed ore will be drawn from the crushed ore bin below the crusher by an apron feeder and discharged onto the secondary crushing circuit conveyor. This conveyor discharges into the Secondary screen feed bin which also serves to split the feed to two parallel vibrating screens.

Screen undersize will be conveyed to the crushed ore stockpile, which provides surge capacity between the crushing circuit and the remainder of the plant (HPGR/ ball mill/ flotation). Screen oversize will be conveyed to a secondary crusher feed bin which also serves to split the feed to two parallel secondary crushers. Crusher product will be conveyed back to the screens feed bin in a closed loop configuration.

The crushed ore stockpile will be partially enclosed in a dome shelter and will house a generally conical shaped ore pile with an average angle of repose of 35 degrees. The stockpile will be 35 m high and will have 28,400 t of live capacity, providing up to 28 hours of buffer between the mining and processing operations.

Figure 18-2: Primary, Secondary Crushing and Crushed Ore Stockpile



Source: Lycopodium (2024)

18.4 Reclaim and High-Pressure Grinding

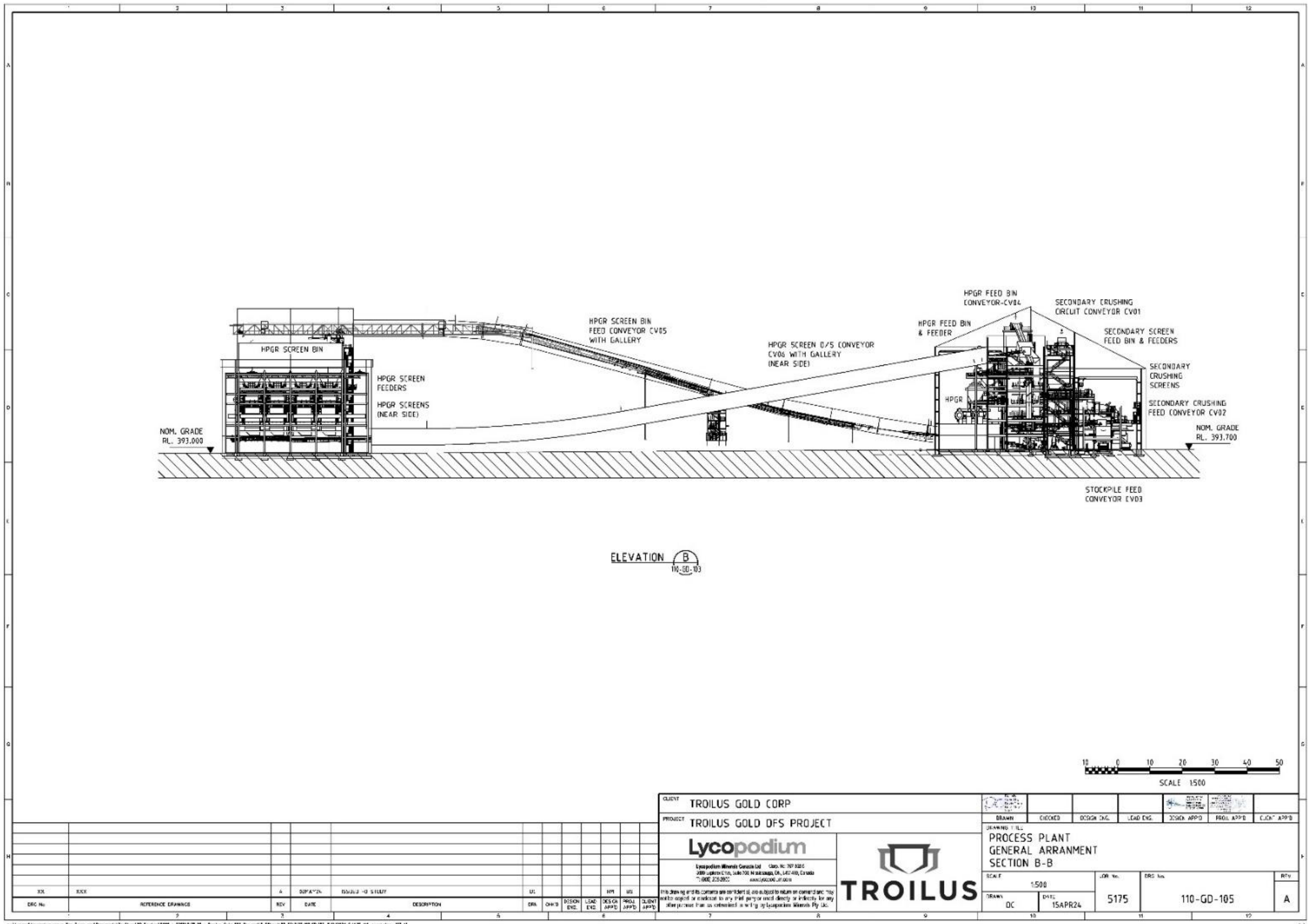
The crushed ore stockpile and reclaim will be located southeast of the process plant. The high-pressure grinding rolls (HPGR) building will be located east of the process plant, in between the primary and secondary crushing facilities and crushed ore stockpile. Figure 18-3 shows the general arrangement and main areas of the reclaim and HPGR building. Additionally, Table 18-3 summarizes the major equipment of these areas.

Crushed ore will be reclaimed via three apron feeders and conveyed to the HPGR feed bin. A belt feeder will draw material from the feed bin and discharge it into the HPGR feed hopper. The HPGR will grind material down to a P80 of -10 mm. This fine material will then be conveyed to a screening circuit that will ensure the oversize report back to HPGR in a closed loop configuration. Undersized material pulped with water discharges directly into pump boxes for milling.

Table 18-3: Reclaim and High Pressure Grinding Major Equipment

Equipment	Size	Power (kW)
Reclaimed ore feeders 1-3	1,214 wet t/h (1,520 mm W x 9.0 m L)	55 each
Reclaimed ore conveyor	5,255 dry t/hr (2,100 mm W x 292 m long x 37 m lift)	900
HPGR feed bin	500 m^3 live capacity	N/A
HPGR	4,380 dry t/hr (2.6 m dia x 2.6 m wide rolls) – HRC 3000	2 x 5,700
HPGR screen feed conveyor	5,255 dry t/hr (2,100 mm W x 245 m long x 36 m lift)	800
HPGR screening surge bin	3,750 m^3 live capacity	N/A
HPGR screens 1-4	4 x Double deck multi slope (3.6 m x 8.5 m) 1,611 m^3/hr slurry	90 each
HPGR screens oversize conveyor	2,414 dry t/hr (1,500 mm W x 248 m long x 28 m lift)	355

Figure 18-3: Reclaim and High-Pressure Grinding



Source: Lycopodium (2024)

18.5 Process Plant

The general arrangement for the process plant is shown in Figure 18-4. The main areas comprising the process plant are:

- milling and classification area
- gravity circuit (future)
- regrind circuit
- rougher, scavenger and cleaner flotation
- concentrate handling
- gold room (future)
- tailings
- water services
- reagents area

The milling area is located approximately 300 metres northwest of the crushed ore stockpile. This area will be an enclosed steel framed building supported on a concrete foundation and fitted with overhead maintenance cranes. The milling building will consist of two parallel lines each containing a grinding mill (8.23 m dia x 13.34 m EGL) and hydrocyclones. The ball mills will be supported on independent concrete foundations.

The flotation building is located adjacent to the mill building on the southwest side. The building consists of a rougher and scavenger flotation circuit; a concentrate regrind circuit, a concentrate thickener, a cleaner 1, 2 and 3 flotation circuit and water services. The building is an enclosed steel framed building supported on a concrete foundation and fitted with overhead maintenance cranes. The building occupies an area of approximately 3,950 m^2 and houses thickener underflow and tailings pumps as well as various other utility water pumps. The 65 m diameter tailings thickener and the process water tank is located outside of the building in a concrete containment area.

The concentrate handling building is an extension of the flotation building on the southeast side. It houses a concentrate filter feed tank, filtrate tank and a concentrate pressure filter. The building will also include a drive through for trucks where concentrate will be loaded via a FEL.

The future combined gravity circuit and gold room building will also be an extension of the flotation building on the northeast side. It will consist of multiple gravity concentrators, multiple shaking tables, a drying oven, diesel smelting furnace and safe/vault.

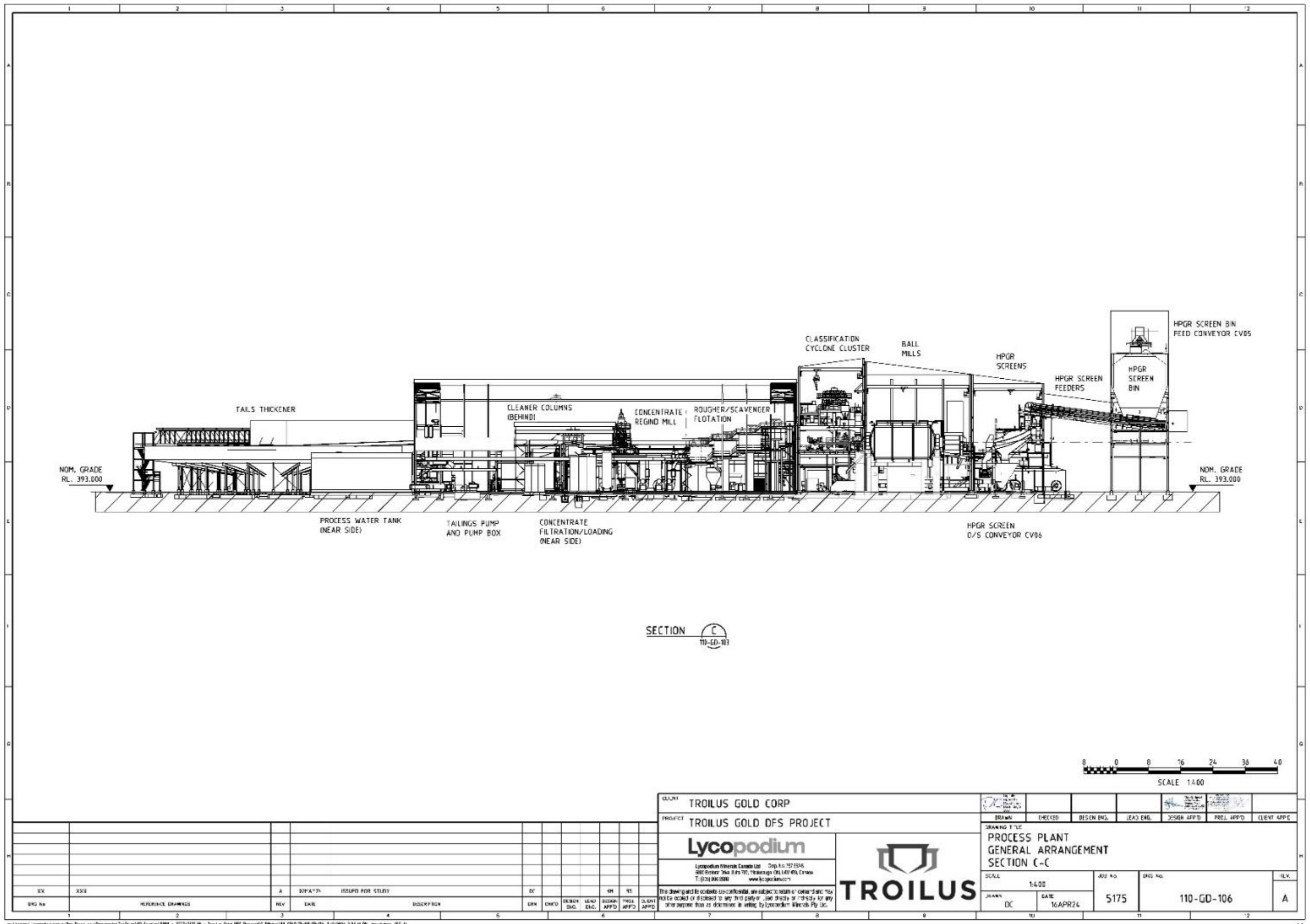
The reagents building will be located northwest of the flotation building. Various reagents storage tanks will be located in separate concrete containment areas. All storage tanks, steel structures, and pumps will be supported on a large concrete slab on grade. This building will be 54 m in length and 20 m wide.

Process plant major equipment are listed in Table 18-4.

Table 18-4: Process Plant Major Equipment

Equipment	Size	Power (kW)
Ball Mill 1 and 2	1,184 dry tph, 8.23 dia x 13.34 m EGL	2 x 9,000 each
Cyclone Cluster No.1/No.2 Feed Pump 1/2	7713 m^3/h @ 36.9 m TDH, 24 x 20	1,800 each
Cyclone Cluster 1 and 2	6,430 m_3/h, 14 off 660 mm – gMax26-H	N/A
Gravity Scalping Screen No. 1 and 2 (Future)	1585 m^3/hr, 4.88 m wide x 9.76 m long	90 each
Gravity Concentrator 1 and 2 (Future)	1000 t/h, 70" diameter / 1780 mm nominal	186.4 each
Rougher/Scavenger Flotation Cells 1-14	65 m3 (200 m^3 equivalent)	110 each
Regrind Mill	170 dry t/hr, P80= 20 microns VRMTM15000-2550	2,500
Tailings Thickener	3,340 m^3/h slurry, 64 m diameter	37

Figure 18-4: Process Plant



Source: Lycopodium (2024)

18.6 Site Buildings

18.6.1 Administration Building

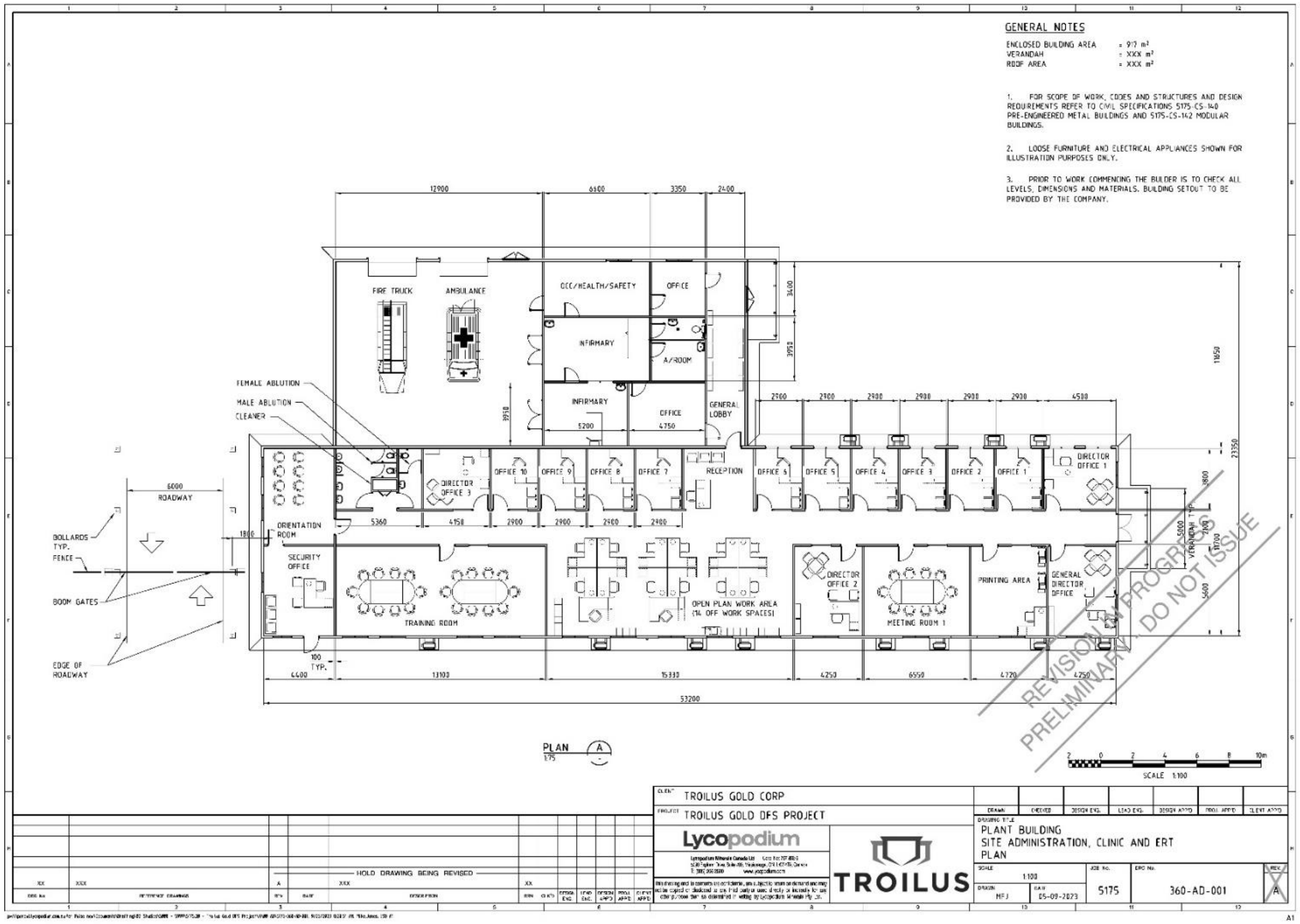
The site administration building will be located southwest of the process plant. This building contains plant and administration offices, technical services offices, meeting rooms, training rooms, reception area, mess room and washrooms. The overall building dimensions are 11.7 m X 37.6 m long x 2.7 high. A portion of the building will serve as the Clinic and ERT building. It will contain the treatment room, ambulance bay, doctor's room, waiting area and restrooms. The administration building will also include a high security building. It will house offices and a search area.

This will be a prefabricated building supported on concrete strip footings. The proposed building layout is shown in Figure 18-5.

18.6.2 Metallurgical and Assay Laboratory

The laboratory will be located east of the flotation plant building. This will be a containerised building with approximate dimensions of 5.5 m X 11.8 m X 4.8 m high. It includes the metallurgical, analytical and fire assay laboratories. The building will also house the sample preparation and receiving areas, instrument room, storage areas and offices. The building will be supported on concrete strip footings.

Figure 18-5: Administration Building



Source: Lycopodium (2024)

18.7 Mine Services Buildings

18.7.1 Mine Truck Shop

The mine truck shop building will be located in the north-east of the ROM pad and primary crushing facility. It will house a mine truck shop, light duty vehicle workshop, oversized/spillover welding bay and a wash bay. The heavy and tracked vehicle maintenance bays will be 4.18 m high. The building will be supported on concrete strip footings.

18.7.2 Mine Warehouse

The mine warehouse will be located adjacent to the mine truck shop on the west side. This building will house a 2000 square meter warehouse and offices at the mezzanine level. This will be a prefabricated building with approximate dimensions of 50 m x 20 m x 8.0 m high.

18.7.3 Fuel Management Facility

The fuel management facility will be located north-east of the mine truck shop / warehouse building. This facility will be designed and supplied by the fuel supply contractor. The delivery trucks will be connected via flexible hoses to the diesel unloading pumps. The diesel will be transferred to bulk fuel storage tanks within a containment bund. The diesel distribution pumps will transfer the fuel to the mine truck refueling slab. The refueling station will be fitted with containment slabs, kerbs, and bunds. Fire protection will be provided by the site's fire protection system.

18.7.4 Camp

At present, there is a small exploration camp on site capable of housing and feeding approximately 60 people.

The camp and related ancillaries will be constructed in two phases. Initially, the camp will be used to accommodate a peak of approximately 530 persons during the construction phase.

The construction/operations camp will be in the same location as the camp used in the original operation from 1996 to 2009. This location offers an already established flat pad.

The construction/operations camp is included as part of a recently obtained quotation received from a local vendor to supply all camp and catering facilities on a lease and per diem basis for the first 5 years. The facilities will include all accommodations, catering, lounges, and a fitness centre.

The camp costs during the construction phase are carried as a capital cost while camp costs during operations are included the G&A costs.

The camp buildings will be prefabricated and transported as 'flat packs' to site for quick assembly on prepared concrete foundations. The structures will be steel framed and insulated interior panels and exterior sheeting panels. Metal doors and windows would be provided. The electrical, plumbing, HVAC, and communications systems for each type of building would be installed at site.

The camp buildings will comprise:

- (11x) 42 Person Jack and Jill style dormitories

- (2x) 36 Person Executive style dormitories
- (3x) Lobby and Reception Units
- (7x) Recreation Centre Units
- (16x) Kitchen and Dining Complex Units
- (1x) Supplementary Freezer / Cooler Unit
- (1x) Laundry Facility Unit
- (1x) Guard House Unit
- (10x) Arctic Corridors
- (1x) Water Treatment Plant
- (1x) Wastewater Treatment Plant
- (1x) Fire Water Pump House

18.8 Power Supply and Electrical

18.8.1 Existing Electrical Infrastructure

The Project is currently serviced by an existing 161 MVA power line that is owned and operated by Hydro Québec. The power line has a capacity of 75 MVA.

At present, the existing HV substation is rated for a maximum nominal 50 MVA.

The HV substation is comprised of the following equipment and infrastructure:

- 161 kV Hydro Québec utility line, single feed
- 1 x 170 kV circuit breaker, live tank, SF6, 3150 A
- 2 x 161 kV disconnects, 600 A each
- 2 x 15 / 20 / 25 MVA (ONAN / ONAF / ONAF) transformers, 161 / 25 kV, fan-equipped, five step off-load tap
- 3 x 34.5 kV circuit breakers, 800 A
- 3 x 25 kV disconnects, 600 A
- 1 x grounding transformer, 26.4 kV, ONAN
- 3 x 25 kV voltage regulators
- control room with 125 VDC battery charger system

There is an existing 25 kV overhead power distribution on site for site services, including:

- temporary camp (500 kVA, 25 / 0.6 kV transformer, ONAN) + (150 kW, 0.6 kV diesel standby generator)
- admin/warehouse buildings (500 kVA, 25/0.6 kV transformer, ONAN)
- old water treatment building (750 kVA, 25/0.6 kV transformer, ONAN)
- tailings pond dewatering pumps (1,000 kVA, 25/0.6 kV transformer, ONAN)

There is also a new 25 kV overhead power line installed to the new water treatment plant.

18.8.2 Main Substation

As mentioned above, the site main substation is fed by an existing 161 kV power line that is owned and operated by Hydro Québec, with power capacity of 75 MVA. Plant full load Maximum Demand (MD) is 84.66 MVA. The line will be upgraded for the additional load.

There are two existing power transformers T1 & T2 each of 15/20/25MVA (ONAN/ONAF/ONAF), 161/25kV, 3 phase both were installed 28 years ago and were not in operation for many years before the rehabilitation of the units was conducted by ABB.

The two transformers will be operated independently (not parallel) and connected to a common bus bar via a bus coupler circuit breaker to enable using one of them instead of the other when needed.

Transformer # 1 will feed 25 kV Overhead Line (OHL) remote load including water management and mine in addition to the site infrastructures.

Transformer # 2 will feed 25kV switchgear of Crushing and Grinding with other loads related to the process of areas 120 & 130.

A new Transformer # 3 of 60/80/100 MVA (ONAN/ONAF/ONAF), 161/25kV, 3ph with protection and control will be installed to feed the remaining process plant.

The following upgrades are planned for the HV substation:

- a new 60/80/100 MVA ONAN/ONAF/ONAF main transformer
- a new 170 kV, circuit breaker, live tank, SF6, 600 A
- a new 34.5 kV, circuit breaker, 3000 A
- a new 34.5 kV, circuit breaker, 1600 A
- protection, control and metering system for new transformer and other equipment
- control alarm panel to be updated to digital relays and control systems
- battery charger system requires new batteries
- new fencing with grounding
- additional insulating stones

The existing 25 kV overhead power distribution will have to be expanded to accommodate the new mine services, various buildings, TSF and remote pump stations.

18.8.3 Power Demand

The calculated maximum power demand (MD) of the entire site is 84.66 MVA and the total connected loads is forecasted at 114. 49 MVA (Refer to Load list TRO-E-LI-0001).

Plant MD consists of two main loads:

- 20.114 MVA of Remote Loads (mine equipment and water management) and infrastructure facilities

- 64.7 MVA of Process Loads consists of two main feeders 14.7 MVA of Crushing areas 120 & 130 and 50 MVA of remaining process plant

The average power demand is summarized in Table 18-5. The average power draw is estimated to be 79,239 kW.

Table 18-5: Power Demand

Area	Installed Power kW	Average Continuous Draw kWh	Annual Power Consumption kWh
120-Feed Preparation	6,766.00	5,650.00	37,119.54
130-Milling and Classification	57,831.00	44,803.00	345,375.00
150-Flotation and Concentrate Handling	9,074.00	6,812.00	52,518.00
170-Goldroom	155.00	124.00	977.00
180-Tailings Handling	2,749.00	1,103.00	8,502.00
210-Reagent	200.00	131.00	1,028.00
220-Water Services	3,112.00	1,547.00	11,929.00
230-Sewage Collection and Treatment	88.00	63.00	488.00
240-Aire Services	1,431.00	1,068.00	8,233.00
250-Fuel Storage and Distribution	112.00	49.00	373.00
330-Water and Sewage	750.00	125.00	966.00
350-Tailing Dam	838.00	453.00	3,490.00
370-Buildings	2,475.00	2,005.00	12,289.00
380-Permanent Accommodation	1,665.00	1,330.00	11,594.00
450-Mine Facilities	103.00	45.00	275.00
Mine Services 87 and J Zone Pits	15,042.00	8,880.00	77,790.00
Other Remote Infrastructure	5,749.00	4,691.00	41,096.00
Total	**108,140.00**	**78,879.00**	**614,042.54**

*Based on the updated Electrical Load List

18.8.4 Process Plant and Ancillary Services Power Supply

Process Plant will be supplied via two MV 25kV switchgears; Crushing and Grinding (areas 120 and 130) will be fed from 342-SG-002 and the remaining process plant will be fed from 342-SG-001. Both switchgears are located inside Electrical House (E-House) 4.

Process loads distributed from suitable MCCs with 2000kVA transformers maintain about 80% of power rating of transformers. Small dry transformers will be used to provide 208-120V to Panel Board for non-process services (lighting, receptacles, and small loads).

18.8.5 Power Distribution Details - Overhead Line

The existing 25 kV overhead power distribution will have to be expanded to accommodate the new mine services, various buildings, TSF and remote pump stations.

A new overhead line is comprised of three main branches of 25kV OHL out of the main substation (refer to OHL drawing TRO-0342-E-SL-0008) to supply the mentioned loads.

Part of the existing OHL will be used as is, another part will be upgraded to higher power as required while in other areas few parts will be removed due to the new load distribution layout. New 25kV OHLs will be installed in different areas according to the new load locations. (Refer to drawing TRO-0342-E-GA-000)

In pit mining equipment will be supplied with 25kV from nearest OHL pole, and Vendors will provide suitable transformers according to the equipment voltages and power. Water Management pumps will be supplied with 25kV power to transformers of 25kV/600V and two of 25/4.16kV as per WSPF Equipment List.

Infrastructure facilities will be supplied with 25kV to transformers of 25kV/600V of different power ratings.

There are no overhead lines within the vicinity of the process plant.

18.8.6 Power Distribution Details - Process Plant

Two 25kV switchgears are the main power distribution for the entire process plant.

Most of the large power drives are fed by 4.16kV feeders from three medium voltage (MV) switchgears (292-SG-001, 292-SG-002 & 292-SG-003), while higher power drives (Ball Mill 1 & 2 each of 18,000kVA VFD and HPGR one 12,000 kVA VFD) will be fed via direct 25kV cable feeders from main switchgear 342-SG-001.

Three cable feeders from 25kV main switchgear will feed three 25/4.16kV power transformers each supplying one 4.16kV switchgears. These MV switchgears are in electrical houses (E-House) located at three areas (120, 130 & 150).

Eleven low voltage 600V Motor Control Centres (MCCs) for process plant loads are supplied via distribution transformers on 2000kVA, 4.16kV/600V. Lighting and small power loads will be fed via 600/208-120V transformers and Panel Boards.

All Power Transformers are outdoor weatherproof oil-immersed type with ONAN/ONAF cooling, while the cooling of Distribution Transformers (5MVA and less) is ONAN or unless otherwise indicated in drawings.

MV and LV cables type Teck90 (unless otherwise indicated) will be used for power distribution and control wiring. Cables are sized based on load rating, current and voltage drop using initial calculations.

Cables will be run underground, direct buried or in raceway as specified, and on tray/ladder protected from mechanical and weather effects.

18.8.7 Voltage Selection

The site voltage system is 161 kV for HV, 25 kV & 5 kV for MV and 600/208-120VAC for LV. Extra Low Voltage ELV is 48 or 24 VDC & VCS for control, instrumentations, and communication.

According to LV & MV Electrical Motors Technical Specifications and Design Criteria, motor voltage shall be as follows:

- motors up to 0.37 kW: 208-120 VAC, 60Hz, 1 or 3 phase, solidly grounded
- motors 0.37 kW up to 200 kW: 575 VAC, 60 Hz, 3 phase, high resistance (10A) grounded
- motors up to 400 kW fed by VFD: 575 VAC, 60 Hz, 3 phase, high resistance (10A) grounded
- motors with a power rating greater than listing above: 5000 VAC, 3 phase or higher voltage (25kV or 11kV) based on power rating and manufacturer design (Based on Design Criteria)

18.8.8 Power Quality

19.5 MVAR is the calculated Reactive Power based on the maximum demand (MD) power of the Site. Power factor shall be corrected to approximately unity by three Power Factor Correction Capacitor (PFCC) banks because of separate load centres. PFCC units are sized based on calculated reactive power as per the Electrical Load List.

One PFCC of 4.5 MVAR will be connected to the outgoing OHL feeder of 25kV in Main Substation for Remote Loads. The second PFCC of 4 MVAR for Crushing areas loads was supplied by (on 342-SG-002) and the third PFCC of 11 MVAR for the remaining loads of process supplied by 342-SG-001.

PFCC banks are provided with a controller for adding capacitors in steps according to the power factor (p.f) at the time. PFCC shall include Harmonic Mitigation system to control the harmonic produced by nonlinear loads.

Voltage might be regulated via On Load Tap Changer (OLTC) for new transformer and voltage regulator for existing transformers.

Hydro HV quality expectation to be within the acceptable ranges without undervoltage dips.

18.8.9 Construction Power Supply

There are numerous existing areas that are currently in operation and supplied from the existing substation by LV transformers that could be used for Construction activities and the camp. Additional power supply could be extended from the 25kV OHL using temporary LV 25KV/600V transformers as required. Prime Generators might be used for works located far from the OHL when needed.

18.8.10 Emergency/Standby Power

Three Standby generators each of 2000kVA will provide power to process essential loads via step up transformers and synchronisation system (switchgear 342-SG-001).

A few remote loads are provided with local small standby generators (Camp and Sewage Treatment Plant). The existing 1.25kVA generator might be used for essential power connected to OHL separately in case needed otherwise. Client to advise.

18.8.11 Switch Rooms and E-Houses

Four main prefabricated (or site constructed) E-Houses will be provided and installed at the process areas. Size and dimensions of E-Houses are estimated depending on load distribution and size of switchgears and MCCs of each area. One small E-House will accommodate low voltage MCC and other services at the Stockpile. Refer to E-Houses Layout.

18.8.12 Control System and Instrumentation

The Plant Control System (PCS) will be configured as a three-tiered pyramid network. The lowest tier will comprise field instrumentation and control equipment. The middle tier will comprise the process control system hardware. The top tier will comprise the operator interface hardware.

The PCS will be programmed in accordance with the project P&IDs and Control Philosophy. The system will be configured such that modifications, troubleshooting and fault finding will be able to be carried out by maintenance personnel without extensive training.

The PCS will consist of a PLC network and SCADA system as per the process sections.

A PLC will generally be installed for each main area as required. Each PLC will be a standalone unit complete with CPU. Field located remote I/O panels will be utilised and connected to the relevant area CPU. All PLCs and remote I/O panels within the Plant will be linked on a common Ethernet communications network, where practicable.

The SCADA system will consist of several Operator Stations running proprietary software, which will provide the operators with a graphical representation of the plant. Each Operator Station will consist of a standard desktop computer, dual colour monitors, keyboard, and mouse. (Refer to Design Criteria)

18.9 Logistics

During construction and operation phases equipment and operating supplies will be received on site daily. These include local deliveries from various parts of the world including the United States and overseas.

Most of the equipment/operating supplies will be delivered in containers or as break-bulk loads delivered directly to the site. All transportation to the site will be verified in Chibougamau before being dispatched to the site ensuring that communication protocols are understood and maintained whilst travelling between Chibougamau and site.

During construction, a marshalling area will be set up in Chibougamau to check inventory, store equipment as necessary, and co-ordinate the delivery to site.

There is a train transshipment dock In Chibougamau and a truck transshipment dock at site.

There is radio communication from town to site through FM and CB channels.

There is a heavy machinery repair shop in Chibougamau and at site. Caterpillar, and another company, have on-the-road units for emergency repairs.

Both the town and the site have lodging capacity capable to answer of the demand.

18.10 Fuel Supply

Diesel will be sourced from a local Québec fuel supplier and delivered to site based on dyed diesel.

Bulk fuel supply will be provided by an on-site fuel storage facility and will store diesel for mine trucks and light vehicles. Bulk lubrication and diesel fuel dispensing are provided for the mine trucks and light vehicles. A local vendor will supply the diesel fuel.

18.11 Potable Water

Fresh water from Lake A will be treated in a water treatment plant (WTP) and distributed to various locations on site. The largest potable water user during operations will be the permanent camp.

18.12 Sewage and Solid Waste Management

18.12.1 Sewage Treatment

Effluent from all water fixtures from the permanent camp, administrative buildings, process plant and mine facilities areas will be pumped to a common sewage treatment plant (STP). Treated effluent will be discharged to the environment. The STP sludge will be suitable for direct landfill burial.

18.12.2 Solid Wastes

Wastes will be sorted, reused, or recycled as much as possible. Waste have cannot be recycled will either be incinerated or buried in the landfill. Waste lubricating oils will be removed and disposed of by the fuel provider. Dangerous or hazardous waste will be collected and stored briefly before being transferred to a suitable permitted facility, either onsite or offsite depending on the specific materials and requirements.

18.13 Plant and Mine Facilities

18.13.1 General

Site buildings will be 'fit-for-purpose' industrial modular-type structures.

18.13.2 Plant Area

The following buildings and facilities will be provided for the plant area:

- gatehouse with first aid clinic, ambulance, and fire truck
- administrative building to house the general manager, process, and mine services staff
- plant workshops
- assay and process plant laboratories

18.13.3 Mine Services

The following buildings and facilities will be provided for the mine facilities area:

- mine truck shop and warehouse
- fuel storage area for refuelling mine trucks and light vehicles
- explosive compound to safely store blasting material
- explosive mix plant
- truck washdown facility

18.14 Site Security

The main entrance gatehouse located in the combined admin and ERT building will be staffed 24/7. All employees and visitors will sign in and out, and an active log of all site individuals will be maintained.

In future Doré will be secured in a locked area with restricted access. Doré will be shipped from via a helicopter to an armoured carrier in Chibougamau. The armoured carrier will transport the doré to a refinery in Québec.

18.15 Site Wide Water Management

Troilus commissioned WSP to prepare the feasibility site-wide water management plan for the Project. The water management plan considered the site development sequence using the available mine plans.

The mine water management plan includes the following main components:

- Natural water management - this will be achieved by diverting natural streams around the disturbed footprint.
- Contact water management - contact water refers to water that will be collected within the development footprint and that may contact waste rock, mine tailings, access roads, or other site infrastructure.
- Pit dewatering and pit water management - this item was outside WSP's scope and is addressed in Chapter 21. The water management plan did account for the impact of pit water pumping to other facilities and its role in the overall site-wide water balance.
- Tailings water management - this includes collecting water consolidated from the tailings and returning it to the ore processing plant.
- Fresh water supply - this item was outside WSP's scope and is addressed in Sections 17.13 and 18.11.
- Surface water monitoring.

WSP's work scope involved the preparation of plans, construction sequencing, and feasibility level design for the items listed above, minus the indicated exceptions, as well as the estimation of construction quantities for the designed water management structures. Results are provided in the following sections. The preparation of a site-wide, operational water balance and conceptual closure considerations were also included in the work scope.

Contact water quality assessment was not in WSP's scope. Troilus indicated the following:

- Historical monitoring of runoff and seepage water quality from the existing waste rock dumps and the tailings storage facility (TSF), which were developed during the previous Site operation ending in 2009, indicated no water quality issues that would prevent the water's release to the environment.
- As an exception to the previous statement, there are elevated iron concentrations in the seepage water collection at the toe of the TSF, which Troilus has been managing by returning small volumes of water to the TSF water pond. Also, certain current effluents exceed specific

surface water quality criteria for the protection of aquatic life (chronic effect) for aluminum, cadmium, copper, and zinc, but the concentrations remain below applicable effluent guidelines.

- Based on their understanding that future waste rock and tailings characteristics will resemble those of the previous operation, Troilus instructed WSP to consider that contact water quality would comply with release water quality guidelines. Total suspended solids (TSS) concentration was the only water quality parameter that WSP considered in the preparation of the water management plan.

18.15.1 Water Management Objectives

The objectives of the water management system are to:

- Facilitate efficient mine and plant construction, operation, and closure by:
  - reducing onsite water handling requirements
  - providing a secure supply of water to meet the project water requirements for production
- Reduce effects on downstream receiving waterbodies by:
  - optimizing the diversion of natural flows around the development area
  - managing contact water to meet the regulatory requirements for the water releases

The above-mentioned water management objectives will be achieved through the design, construction, and operation of the following water management facilities:

- stream diversion channels and a stream diversion dam to divert runoff from upstream natural watersheds around the Project development area
- drainage ditches, sumps, pumps, and sedimentation ponds to manage the contact water and to intercept non-diverted natural runoff and groundwater seepage so that the water is of acceptable quality prior to release to the receiving environment
- wells, pumps, and pipelines to dewater the pits
- water pond within the TSF and TSF emergency spillway
- pumps and pipelines from Lake A to supply the freshwater demand at the plant
- surface water and groundwater monitoring stations to collect water quantity and quality data for regulatory compliance monitoring

WSP (2024c) includes the feasibility level design of the discussed water management structures. Exceptions are the pit dewatering systems, which were outside WSP's scope, and the monitoring systems, which were proposed on a conceptual level only.

The following paragraphs present high-level elements of the Troilus Project Operational Site-Wide Water Management Plan (WSP, 2024c).

18.15.2 Water Management Design Criteria

Table 18-6 presents the proposed water management design criteria. They are based primarily on the 2012 provincial government's Directive 019 for the mining industry and on the Canadian Dam

Association's dam safety recommendations. As indicated in Section 18.16.1, WSP assumed that TSS concentration was the only water quality parameter to require treatment.

Table 18-6: Water Management Structures Design Criteria

Structure	Design Criteria	Source and Comments
Bibou Creek Diversion	Manage the 100-year peak design flood without uncontrolled overtopping or significant structural damage	Directive 019 (MELCCFP, 2012)
	Allow 0.3 m freeboard between the top of the banks and the peak design flood water level	General practice to account for design and construction uncertainties, ice effects, debris accumulations, and waves
	Provide fish passage during the 7Q10 October low flow event and the 14Q2 spring freshet high flow event, with consideration of migration of walleye in the spring and passage for burbot during average annual open-water conditions	WSP based on current understanding of fish and fish habitat conditions
	Side slopes: 2.5H:1V in till 1H:1V in bedrock	Typical side slope for moderately competent soil and for excavation in bedrock. For excavation in bedrock, the slope includes the potential requirement for safety benches
	1 m minimum base width and 1 m minimum depth, with the exception of the low flow (fish passage) channel	Practical considerations for construction
	Provide adequate erosion protection riprap for all structures, with a maximum rock size of 500 mm	Limit structural damage due to erosion and mobilization of TSS, and make use of available on-site materials sourced from sorted blast rock
Collection ditches and secondary diversions	Manage the 100-year peak design flood without uncontrolled overtopping or significant structural damage	Directive 019 (MELCCFP, 2012).
	Allow 0.3 m freeboard between the top of the banks and the peak design flood water level	General practice to account for design and construction uncertainties, ice effects, debris accumulations, and waves
	Side slopes: 3H:1V in till 0.5H:1V in bedrock	Typical side slope for moderately competent soil and for excavation in bedrock
	1 m minimum base width and 1 m minimum depth	Practical considerations for construction
	Provide adequate erosion protection riprap for all structures, with a maximum rock size of 500 mm	Limit structural damage due to erosion and mobilization of TSS, and make use of available on-site materials sourced from sorted blast rock
Sumps and sedimentation ponds	0.5 m minimum depth allowance for sediment accumulation	Professional judgment to accommodate uncertainty in design, construction, and

Structure	Design Criteria	Source and Comments
	0.5 m minimum freeboard above design water level	operation and to allow for practical frequency of pond cleaning. For some specific structures, sediment accumulation and freeboard allowances may be reduced.
	For sedimentation ponds, allow sufficient residence time for the settlement of a reference 10-micron particle during the design flood event	Directive 019 (MELCCFP, 2012) with respect to the discharge guidelines and the British Columbia sedimentation pond design manual (British Columbia Ministry of Environment 2015)
	3H:1V side slopes	Typical side slope for moderately competent soil
Pumping systems	Designed to operating flowrate with conservative pump selection	Professional experience and judgment to accommodate uncertainty in design, operation, and equipment performance
	Head loss safety factor of 10%	
	Pump efficiency of 60%	
	Factor of safety on installed pumping power of 15%	
TSF water pond and water treatment capacity	Manage the Directive 019 “project flood” associated with the 1,000-year 24-hour rainfall superimposed with the melt over 30 days of the 100-year snowpack Maintain a freeboard, relative to the crest of the dyke, of minimum 1.5 m during the “project flood”	Directive 019 (MELCCFP, 2012)
TSF emergency spillway	Convey the PMF peak design flow with a minimum freeboard, relative to the crest of the dyke, equivalent to the 100-year extreme wind effect (wind set-up and run-up) Design the spillway channel to be non-erodible during the PMF event unless the channel erosion would not impact the dam (e.g., at sufficient distance downstream of the dam)	Directive 019 (MELCCFP, 2012) and Canadian Dam Association (2013, 2019) recommendations
General	Consider the impact of climate change on design storm events by using available median projections for the planned project LoM	Current standards including Global Industry Standard on Tailings Management (Global Tailings Review [GTR] 2020) and Strategic Assessment of Climate Change (ECCC 2020)
Road Crossings	Manage the 100-year peak design flow	Proposed for FS-level design
	2H:1V slopes on culvert ends	
	0.3 m maximum backwater	
	Assume 5 m of erosion protection is required upstream and downstream of crossing	

Notes: TSF = tailings storage facility; TSS = total suspended solids; PMF = probable maximum flood; LoM = life-of-mine; FS = feasibility study.

18.15.3 Hydrology Settings

Section 20.3.2 presents the general setting of the project area, including a description of the regional hydrographic network and regional watersheds. A baseline climate analysis was completed to support the feasibility water management study:

- Environment and Climate Change Canada historical records at climate stations around the Chibougamau and Chapais towns, 140 km to the south of the site, provide a reasonable basis for extracting climate statistics as input to the design of the Project surface water management structures and in support of the site's water balance. When relevant, the data were adjusted to the site using Ouranos (2024) regional climate maps.
- Based on the analysis described above and, specifically, the 1982 to 2023 historical records at the Chibougamau-Chapais A station, average climate conditions at the site are estimated as follows:
  - The mean annual temperature is -1.0°C, varying between 16.6°C in July and -20.6°C in January (mean monthly daily temperatures). Temperatures stay below zero, on average, between mid-October to mid-April.
  - The mean annual precipitation is 960 mm. July and September are the wettest months (130- and 120-mm mean precipitation, respectively), and January and February are the driest months (42- and 38-mm mean precipitation, respectively).
- Based on the same 1982 to 2023 Chibougamau-Chapais A precipitation records, the 10-year wet and dry annual precipitation was estimated to be 1,107 mm (wet) and 821 mm (dry).
- Extreme rainfall and rain on snow intensities, and maximum snow water cover amounts were estimated using historical data from a combination of stations around Chibougamau and Chapais. Table 18-7 below provides key statistics. For a 100-year return period, WSP estimated a 90 mm 24-hour rainfall, 70 mm 24-hour rain on snow, and a 375 mm snow water equivalent snow cover.

Table 18-7: Extreme Rainfall, Rain-on-Snow, and Maximum Snow Cover Based on Historical Climate Records

Return Period (years)	Summer-Fall Extreme 24-hour Rainfall (mm)	Spring Rain on Snow Extreme 24-hour Event (mm)	Maximum Snow Cover (mm Snow Water Equivalent)
2	456	36	225
5	59	45	270
10	68	51	298
20	75	59	331
50	84	64	354
100	90	70	375
1,000	109	87	440
2,000	114	92	457
10,000	125	103	494
PMP/PMS	335	180 (spring rainfall only)	750

Note: See the Site-Wide Water Management Plan report for a discussion of the data sources and the methodology used to extract these statistics. Presented data are before adjustment for climate change. PMP = Probable Maximum Precipitation. PMS = Probable Maximum Snow Accumulation.

- Historical monthly means for lake evaporation and potential evapotranspiration for the Site were estimated based on 1982 to 2023 climate records. Maximum monthly values were estimated to 116 mm and 144 mm for July for lake evaporation and potential evapotranspiration, respectively. Mean annual values were estimated to 515 mm and 661 mm, respectively.
- An analysis of historical wind data concluded with an estimate of 2-year hourly extreme wind velocities between 23.3 km/h (wind from the East) to 42.6 km/h (wind from the South or Southwest) and of 100-year hourly velocities between 48.3 km/h (wind from the North) to 69.6 km/h (wind from the South).

Based on median climate change projections for a moderate greenhouse gas emission scenario presented by Ouranos (2024), climate at the site is expected to be warmer (1.4°C increase in the annual mean) and wetter (6% increase in the annual precipitation) for the 2021-to-2050-time horizon relative to the 1991 to 2020 period. The changes are larger during the winter than during the summer.

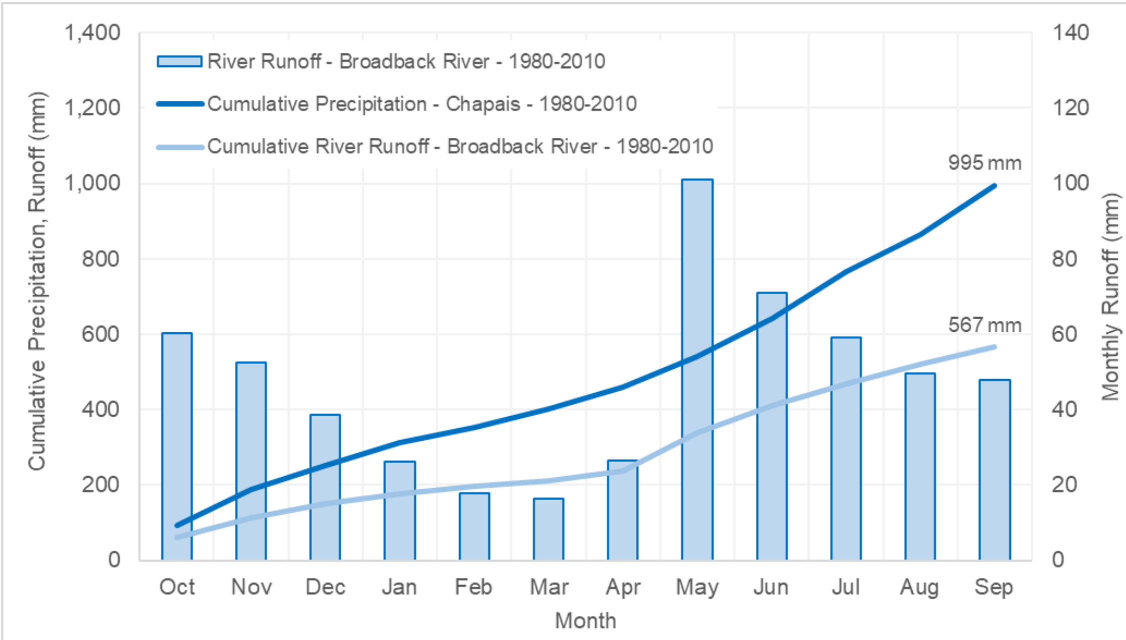
Climate change impact was accounted in the feasibility study water management plan by:

- Increasing extreme, short-term rainfall and rain on snow intensities by 18% as a reasonable adjustment in support of the water management plan for the Troilus Project feasibility study. The value corresponds approximately to the 50th percentile of the projections for a high greenhouse gas emissions scenario. No adjustment was made for snow cover amounts.
- Extracting 78 different climate change projections for the Site region from public databases and simulating the Project's water balance under continuous time series considering these projections.

To characterize runoff from natural, undisturbed areas, the historical streamflow records from the regional Broadback River station were analysed. The mean annual runoff for a large natural area over the 1980 to 2010 period was estimated to be 567 m. Mean monthly values varied between 16 mm in March, before the freshet, and 101 mm in May, the main freshet month as displayed in Figure 18-6.

Based on professional judgment, mean annual runoff coefficients for disturbed areas on the Site were estimated to vary between 60% for waste rock dumps, 68% for cleared areas of the mine site, 77% for the tailings beach, and 86% for the open pit areas. These annual runoff coefficients are summary results of daily calculations that include evapotranspiration and infiltration losses and that account for antecedent moisture conditions near the surface. The daily effective runoff coefficients vary with the intensity of the rain and snowmelt event.

Figure 18-6: 1980 to 2010 Mean River Runoff at the Broadback River Station



Source: WSP (2024)

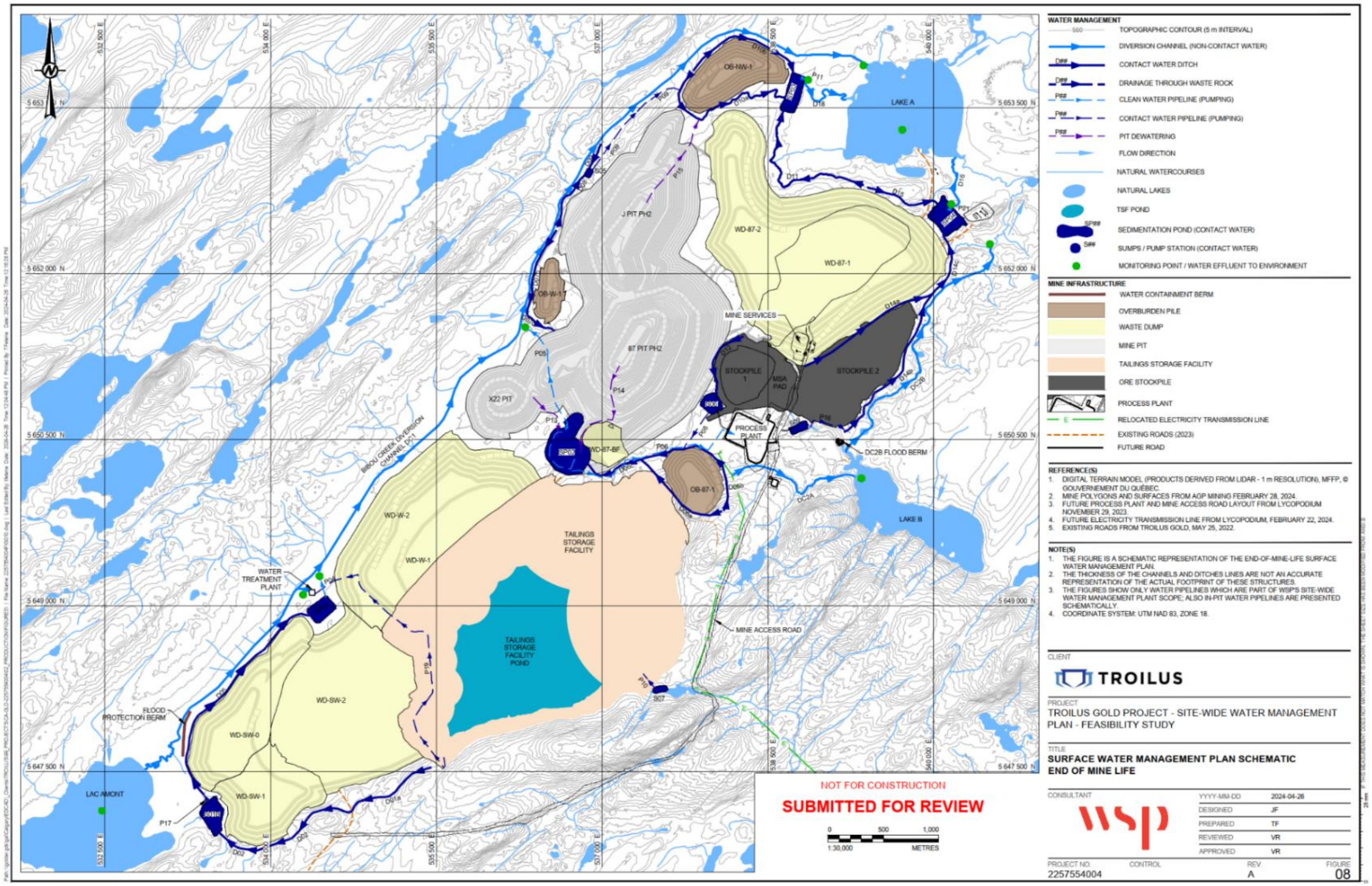
For short-term flood events and small drainage areas, the Soil Conservation Service curve number model was applied to simulate peak design flows. The applied curve number values for the different land types were 60 for natural, wooded areas, 71 for natural, vegetated but not wooded areas, 72 for waste rock dumps and rock-covered dikes, 86 for cleared area, 91 for overburden piles, 94 for the tailings beach, and 93 for the open pits.

18.15.4 Conceptual Water Management Plan

WSP (2024c) prepared conceptual water management plan layouts for the existing (2023) condition, for Project development years -2, -1, 1, 2, 6, 8, and life-of-mine, and for closure conditions. Figure 18-7 presents the proposed conceptual water management plan for the ultimate, life-of-mine site layout, and mine development plan:

- Stream diversion is required to divert the natural Bibou Creek flow around the site to Lake A, downstream of the site. Southwest Pit, the J Waste Dump, and the widening of the 87 and J pits block the alignment of Bibou Creek and its current diversion channel. The 9.7 km long DC1 Main Diversion Channel replaces the existing Bibou Creek and is designed to convey the 100-year flood flow from its 21.4 km^2 drainage area without overtopping or erosion and to provide fish passage between the two lakes. The upstream 5.2 km of the channel is designed to be maintained at closure and will be integrated into the project closure plan.

Figure 18-7: Conceptual Water Management Plan for the Ultimate Site Layout



Source WSP (2024)

- The secondary diversion system, DC2, has a total length of 4 km and conveys water around the east perimeter of the site between Lac B and Lake A. There are two separate channels in the DC2 system. The upstream channel, DC2a, diverts runoff away from the plant site and conveys it to Lac B. The downstream channel, DC2b, conveys flow from the Lac B watershed around the east side of the future stockpile and connects to an existing channel, which flows to Lake A. It is designed to convey the 100-year flood flow without overtopping or erosion. Based on available information, WSP assumed that there is currently only limited fish passage connectivity between Lac B and Lake A. At the Feasibility Study level, WSP did not consider fish passage for this diversion system. Further details will need to be verified during the next engineering phase.
- Four sedimentation ponds - SP1, SP2, SP3, and SP4 - are collecting contact runoff from the site and pit dewatering flows. The drainage areas reporting to the four ponds are 3.2, 9.6, 3.7, and 2.7 km^2, respectively (these areas include both gravity and pumping drainage areas from sumps and pits). The ponds are providing sufficient residence time to settle suspended solids before the water is pumped to the environment. Water from SP1 and SP2 is pumped to the DC1 Main Diversion Channel, while water from SP3 and SP4 are pumped to existing tributaries of Lake A. SP1, SP3, and SP4 are to be built by excavation into existing ground. SP2 is created by repurposing the southern extremity of the 87 Pit; this area is mined at the very beginning of the site development. The total estimated required live storage volume for the four ponds is around 421,000 m^3.
- A network of collecting ditches, which includes 22 ditches with a total length of 18.9 km, and seven pump sumps – S01/S01b, S02, S03, S05, S06, S07, and S09 collect contact water and convey it to the sedimentation ponds or, in the case of S02, S03, and S07, to the TSF Pond. The total estimated required live storage volume for the seven sumps is around 186,000 m^3. One of the sumps (S01) is being relocated during the operations to S01b; counting for the relocation, the total required live storage volume over eight sumps is around 261,000 m^3.
- TSF beach runoff, tailings bleed water, and runoff from the mountainous slope southeast of the TSF are collected in the TSF Pond; the drainage area reporting to the pond is 5.8 km^2. From there, water is reclaimed as process water at the ore processing plant. The pond excess water is treated by a flocculation treatment plant and discharged to the environment via the DC1 diversion channel.
- Overall, the Site remains in the Lake A drainage area. Runoff from 0.43 km^2 is being diverted from neighbouring watersheds into the Lake A watershed due to the alignments of the Main Diversion Channel and the contact water ditches.

Table 18-8 summarizes the site main drainage areas.

Table 18-8: Water Management Structures Drainage Areas

Structure	Drainage Area – Gravity or by Pumping
DC1 Main Diversion Channel	21.4 km^2 (the direct gravity drainage area). Pumped flows from Sedimentation Pond 1 and Sedimentation Pond 2 are also directed to the Main Diversion Channel. They represent 6.2 km^2 of additional drainage area.
DC2a Diversion Channel	1.3 km^2
DC2b Diversion Channel	5.5 km^2
SW Pit (before backfilling)	1.1 km^2
87 Pit	2.3 km^2
J-Zone Pit	2.0 km^2
X22 Pit	1.3 km^2
SP1 Sedimentation Pond	3.2 km^2 (includes SW Pit area prior to backfilling of the pit)
SP2 Sedimentation Pond	9.6 km^2 (includes 87 Pit and X22 Pit areas)
SP3 Sedimentation Pond	3.7 km^2 (includes J-Zone Pit area)
SP4 Sedimentation Pond	2.7 km^2
TSF Pond	5.8 km^2

The described water management plan is valid for the ultimate site layout. The construction of the water management structures will be staged to accommodate the development schedule. Table 18-9 presents the proposed construction staging, which will be confirmed during the next engineering phase.

Table 18-9: Water Management Structures Construction Staging

Operation Year	Proposed Construction
Year -3	• Construction begins on the Bibou Creek diversion, secondary diversions, and sedimentation ponds to ensure they are commissioned and online for the Year -2 freshet.
Year -2	• Existing Bibou Creek diversion is routed around the first phase of the 87 Pit expansion through the bypass diversion channel (DCB) temporary diversion channel. • Construction of the east diversion system (DC2a and DC2b) to divert the Lac B watershed around the stockpile footprints and back to the Lake A inlet channels. • Development of contact water management infrastructure in the north-east end of the site, including SP04 (pumping to Lake A) and associated contact water ditches around the stockpiles. • Development of the first phase of the 87 Pit expansion includes excavation of the basin for in-pit sedimentation pond SP02, which pumps release water to DC1. Construction of the pumping station within the pond is completed, which will direct water to both the main diversion and to the plant for process water. Ditches upstream of the 87 and J pits are also constructed and drain to the Sedimentation Pond SP02. • Plant site expansion and construction, including sump S06, which collects stockpile and plant site runoff and pumps it to SP02, and S09, which pumps contact water (including some plant site runoff) to SP04.
Year -1	• Construction of new stream diversion “DC1” (also called “Main Diversion Channel”), which conveys flows from Lake Amont to Lake A and intercepts runoff from the west side of the valley. • Construction of SP01, which will pump release water to DC1, begins in Q3/Q4 to ensure it is online for the freshet the following year. The existing diversion downstream of the construction footprint continues to collect local runoff, which is conveyed to SP02. • Development of water management on the south side of the Tailing Storage Facility as it expands, including S02 and S03 (both pumping contact water to the TSF) and associated contact water ditches.

Operation Year	Proposed Construction
Year 1	• Construction of water management infrastructure around the Southwest Pit. • Construction of sedimentation pond SP01 is completed to collect runoff and contact water from the Southwest Pit and associated overburden and waste rock dumps. Contact water ditches upstream of SP01 are also completed. • Construction of sump S01 in the footprint of the existing Lac C2 and associated contact water ditches. This sump collects runoff from an overburden dump and from local drainage areas and pumps it to SP01.
Year 2	• Expansion of 87 Pit and Southwest Pit, and associated waste rock and overburden dumps • Additional contact water ditches are constructed adjacent to a new overburden dump near DC1 to intercept runoff from the dump. Runoff is diverted to 87 Pit and pumped to SP02. • Installation of a new water treatment plant for the TSF Pond surplus water. The new plant's location is west of the SP01, near the DC1 channel.
Year 6	• Expansion and development of 87 Pit, J Pit, and the Southwest Pit, and associated waste dumps. • Construction of sedimentation pond SP03, which pumps release water to Lake A, and associated contact water ditches to manage runoff from the expanded 87 Waste Dump and the NW Overburden Dump. • Construction of sump S01b in the existing Lac C footprint to replace the S01 sump which is removed with the expansion of the Southwest Waste Dump. Construction or realignment of associated contact water ditches. Pumping infrastructure is moved from S01 to S01b, which now pumps contact water to SP01 via ditch D05.
Year 8	• Continued expansion of pits and associated waste rock and overburden dumps. • Expansion of the TSF required the construction of the TSF East Dyke across the saddle to the east of the existing TSF. • Construction of sump S07 to manage runoff into the existing water body east of the TSF. S07 pumps contact water into the TSF.

Operation Year	Proposed Construction
Year 22	• The site footprint is fully developed. Mining ends in Year 21, and processing of stockpiled ore is completed in Year 22. • The Southwest Waste Dump is expanded to cover the southwest pit and is integrated with the TSF dyke, covering S02, S03, and associated ditches.
Closure	• Dumps and tailings are reprofiled and covered with reclamation material. • The combined 87 Pit, X22 Pit, and J Pit are allowed to fill using groundwater inflows and local runoff over a transition period. During this time, the existing diversion is maintained. • Once the pit lake is full, the DC1 diversion channel is routed through the lake, and the downstream portion is decommissioned and reclaimed. • The pit lake is connected to the downstream existing channel to maintain a connection to Lake A. • Most collector ditches are decommissioned. Sumps and ponds are left in place as small lakes to collect natural runoff and groundwater inflows.

A closure drainage plan was not prepared as part of the FS. General closure drainage considerations include the following:

- Cover and reprofile the waste dumps, overburden piles, and the tailings beach to encourage efficient and erosion-resistant drainage paths.
- For the TSF:
  - Maintain a permanent pond, whose level would be controlled by an outlet channel draining north across the waste dump near the TSF to the future pit lake. The permanent TSF Pond water level would be lowered relative to the operational conditions such that the long-term freeboard could accommodate extreme scenarios without risk to the safety of the TSF dykes.
  - Place granular material within the TSF pond footprint to control suspended solids and provide dust control as the pond is lowered.
- Maintain the upstream 5.2 kilometres of the Bibou Creek Main Diversion Channel as a permanent stream. The channel is designed such that aquatic habitat is expected to be established in this section of the channel during operations. Channel design elements to promote a smooth transition to closure include:
- Avoid the use of synthetic materials (geotextiles, geomembranes) in the permanent section of the channel. Preferentially select natural materials (e.g., granular filters, low permeability, soils) which are more robust over the long term.
- Design of channel geometry such that extreme flood events will be either managed within the channel or overtopping would not lead to a loss of long-term functionality.

- Promote establishment of aquatic habitat through features such and low flow channels, integration of aquatic habitat features (e.g., riffles, pools, appropriate substrate), which are observed in regional natural water courses, and management of higher flows to preserve established habitat.
- Decommission the downstream 4.5 km of the Bibou Creek diversion channel DC1.
- Route the Bibou Creek diversion channel DC1 through the future pit lake. A channel approximately 0.6 km long will be constructed to connect the diversion to the pit lake.
- Allow natural runoff and groundwater inflows to flood the pits and the excavated sumps and ponds to create closure waterbodies. A closure transition period will be required to allow the pit lake to fill naturally while maintaining fish passage and preserving established habitat in the diversion.
- Decommission most collector ditches and construct a new network of closure drainage features. The closure drainage features will, among others, include specific aquatic habitat features.

18.15.5 DC1 Main Diversion Channel

The DC1 Main Diversion Channel was designed using a 1D hydraulic numerical model. Design flows were calculated using both single flood event modelling and Québec regional hydrologic data as described by WSP (2024c).

The diversion includes a 0.5 km long section that is to be constructed in fill, above the natural terrain elevation. General waste rock is planned as fill material. The channel in this section will be lined with natural low-permeability material and bituminous geomembrane to limit exfiltration.

The design of the Bibou Creek diversion (DC1) is summarized in Table 18-10 and Figure 18-3. Measures to accommodate fish passage include:

- Using a meandering low flow channel design for the upstream portion of the channel (segments 1 through 8 in Table 18-6 and Figure 18-3). A sinuous, low flow channel is located within a larger flood flow channel. The flood flow channel will have boulders placed in strategic locations to create shelter during periods of high flow. Woody debris will also be placed approximately every 20 m to increase water depth in the low flow channel. All erosion protection is blended with sandy gravel to serve as interstitial fill in the rock voids, which will increase flow depth in the low flow channel during periods of lower flow and will improve the quality of fish habitat substrate.
- Including small weirs for flat reaches (segments 9 through 12, 14, and 16 in Table 18-6 and Figure 18-8). This is implemented in the operational portion of the channel (that is, the segments that are planned to be dismantled for closure and where active maintenance is available), where the slope of the channel is shallow. The channel is lined with gravel and cobble to act as a natural analogue substrate for fish passage. A weir is placed every 100 m to create a backwater effect and increase the water depth during periods of low flow.
- Designing steep segments (13 and 15 in Table 18-19 and Figure 18-11) as a series of rock ramps with a 3.5 % slope and pools to achieve a compound slope of 1.7%. Rock ramps are nature-like fishways which provide fish passage and aquatic habitat through simulation of a natural stream

environment. Rocks and boulders are placed in strategic locations to decrease velocity and increase flow depth. The placement of the rocks and boulders mimics a natural stream and allows fish to move from one pool to another. In addition, in this design type, a low flow channel is located within the main channel in the rock ramp sections.

Table 18-10: Diversion Channels Summary Characteristics

Segment Number	Segment Length (m)	100-Year Design Flow (m^3/s)	Longitudinal Slope (%)	Minimum Total Channel Depth [a] (m)	100-Year Design Velocity (m/s)	Width (m) [b]	Erosion Protection Riprap D^{50} (mm) [e]
1	100	17.4	1.0 %	2.12	0.71	11	300
2	600	17.4	0.1%	2.61	0.81	11	100
2	680	17.4	0.1%	2.5	0.97	11	100
4	440	17.4	0.7%	2.20	1.48	11	300
5	300	17.4	0.3%	2.77	0.89	11	100
6	620	25.1	0.1%	2.91	0.92	11	100
7	630	25.1	0.1%	3.24	0.73	11	100
8	1,880	25.6	0.05%	3.53	0.68	11	50
9	920	25.6	0.1%	3.43	1.49	2.5	50 [c]
10	500	25.6	0.05%	4.27	0.80	2	50
11	930	25.6	0.1%	3.40	1.54	2.5	50 [c]
12	390	25.6	0.1%	3.10	2.02	2.5	50 [c]
13	520	25.6	1.7% [d]	2.66	1.14	8	50
14	360	26.2	0.1%	2.56	1.72	2	100
15	430	26.2	1.7% [d]	2.08	1.29	8	50
16	354	26.2	0.1%	2.80	1.06	2	100

Notes:

(a) 100-year design flow depth plus 0.3 m freeboard. Note 1: see details in WSP (2024c) for the location of the segments.

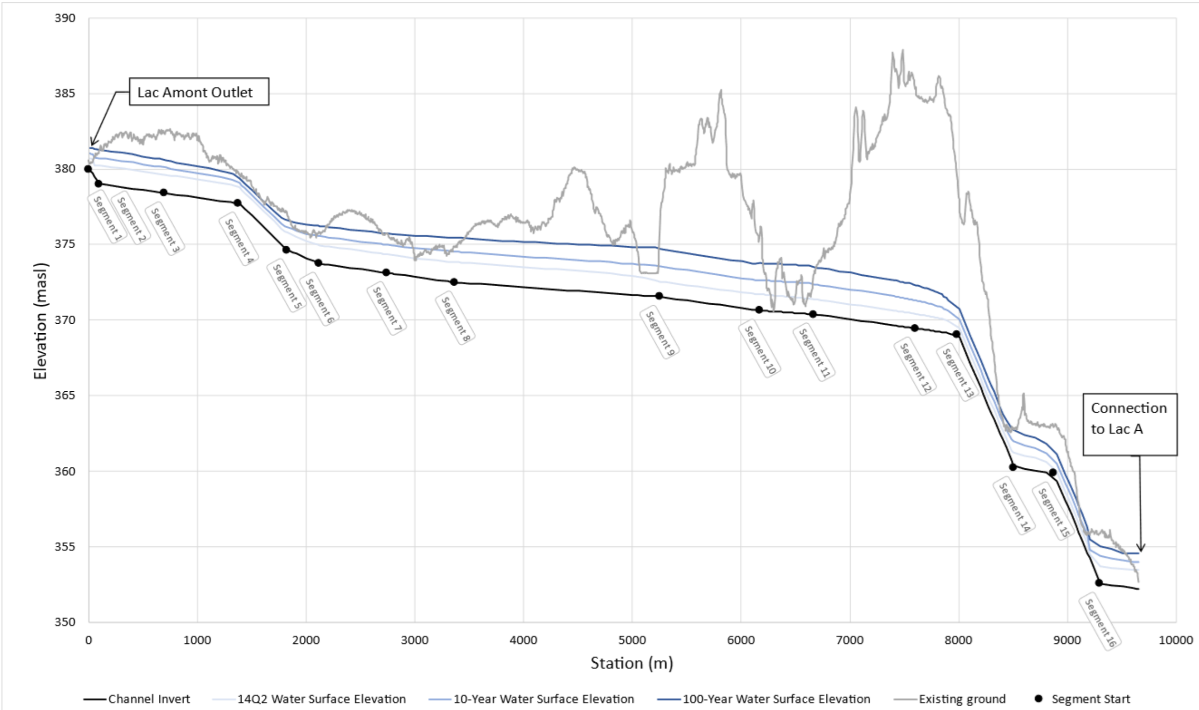
(b) Width of main channel (not low flow channel, where one is included in the design).

(c) Represents substrate placed for fish passage on top of bedrock.

(d) The 1.70% slope represents an average slope through the segment, for hydraulic modelling of flood flows. The compound slope in these segments, which incorporates additional details for fish passage, is represented by the detailed schematic of the rock ramps shown in WSP (2024c).

(e) After the installation of the riprap layer, the channel surface can be smoothed out with a small quantity of gravel that is more favourable for fish passage and aquatic life. Additional specific measures to accommodate fish passage and aquatic life are presented in WSP (2024c).

Figure 18-8: Main Diversion Channel DC1 Invert Profile and Water Surface Elevations



Source: WSP (2024)

18.15.6 Contact Water Management System

There are 18.9 km of planned collector ditches. The ditches will be protected against erosion by riprap. When the topography requires constructing the ditches of fill, a synthetic high-density polyethylene (HDPE) liner is placed to limit exfiltration. The diches were sized using the simulated 100-year design flows and hydraulic analysis.

The pump sumps (eight sumps in total with S01b being a relocation of S01 after the backfilling of the Southwest Pit), are excavated in the existing ground and covered with a 1.0 m thick rock layer for erosion protection, with consideration for groundwater seepage and geotechnical stability. Table 18-11 present high-level sump design details.

Table 18-11: Pumping Sumps Design Details

Sump	Required Live Storage Volume (m^3) (a)	Dead Storage Allowance (m) (b)	Freeboard Above Live Storage (m) (c)	Total Excavation Volume (m^3) (d)	Minimum Excavation Depth for Containment (m) (d)	Total Required Footprint (ha) (e)	Design Pumping Capacity (m^3/s) (a)
S01	74,800	uses existing bathymetry below excavation	0.5	109,000	6.0	5.1	0.25
S01b	74,800	uses existing bathymetry below excavation	0.5	142,500	3.5	5.3	0.25
S02	4,000	0.5	0.5	8,500	5.0	0.6	0.30
S03	15,000	0.5	0.5	63,000	5.5	1.7	0.30
S05	4,000	0.5	0.3	9,700	4.6	0.7	0.20
S06 (f)	53,000	0.5	0.5	103,400	4.6	2.6	0.20
S07	12,000	0.3	0	19,800	5.0	1.1	0.20
S09	23,000	0.5	0.5	73,500	5.0	1.8	0.30

Notes:

(a) Required volume and pumping capacity to manage the 1:100-year flood event without overtopping.

(b) Allowance for sediment accumulation.

(c) Allowance accounting for design, construction, and operation uncertainty.

(d) Including 1.0 m thick rock protection layer and filter layers as required.

(e) Including a 10 m wide band around the excavation perimeter.

(f) The required 100-year live storage volume of Sump S06 is 53,000 m^3. Prior to the construction of Stockpile STK1 the maximum containment volume of S06 containment volume is 38,200 m^3. This volume is sufficient to contain the 10-year design event, which has a peak storage volume of 34,300 m^3. Overtopping of S06 prior to construction of STK1 will send runoff north through existing ditches toward the existing 87 Pit. The construction of STK1 later in Year -2 increases the total containment volume of S06 to approximately 106,000 m^3. To reduce risk of overtopping from the 100-year design flood, if construction of STK1 is delayed beyond Year -2, a 2 m high berm may be constructed on the east side of Sump S06 to increase total containment volume to approximately 71,160 m^3.

Four sedimentation ponds, SP01, SP02, SP03 and SP04, are included in the water management plan. Locations are shown in Figure 18-2. Water collected in the ponds is discharged by pumping to the environment after verification of the water quality conformity with applicable effluent quality requirements. Continuous turbidity sensors are installed on the outlet pipeline, such that discharge is stopped immediately if the TSS concentrations exceed regulatory discharge guidelines.

SP01 and SP02 discharge effluents are to the DC1 Diversion Channel. SP03 and SP04 pump release water to separate Lake A tributaries. For SP02, a pipeline was included in the design to allow pumping to the ore processing plant.

Table 18-12 and the points below summarize the designs for the four ponds:

- The four ponds are to be operated dry to maximize the live storage capacity during a flood event. The pump stations were designed to be installed on floating barges and the intakes are designed to withdraw water from the top of the water column. The water intake to the pumps would be approximately over the top 0.3 m below the water surface. The pond design verified that the residence time is sufficient to allow for a minimum 0.3 m settling depth for the 10 µm critical (small) design particle. As a factor of safety, the pond design added a 4-hour pump stand-bye capacity to account for situations when additional residence time is required during an extreme event.

- SP01, SP03 and SP04 are built by excavation into natural ground. A filter layer of 0.5 m thick sandy gravel over non-woven geotextile is proposed. An active groundwater management underdrain system is not considered based on available data.
- Sedimentation Pond SP02 is created by reserving the southern extremity of the 87 Pit for water management. This small, isolated pit section will be excavated in pre-stripping operations.
- The 100-year flood event criterion was applied for each structure, with an additional 0.5 m for freeboard and 0.5 m for dead storage. For SP01, however, minimal freeboard was included in the design capacity. Should SP01 overflow, the water would follow the terrain profile and seep through the J Waste Dump until the sedimentation pond SP02, which has sufficient capacity to accommodate the excess water from SP01.

Table 18-12: Sedimentation Pond Design Details

Sedimentation Pond	Required Live Storage Volume (m^3) [(a)]	Dead Storage Allowance [(b)]	Freeboard Above Live Storage (m) [(c)]	Total Excavation Volume (m^3) [(c)]	Minimum Excavation Depth for Containment (m) [(d)]	Total Required Footprint (ha) [(e)]	Design Pumping Capacity (m^3/s)
SP01	63,000	0.5	0	192,107	5.5	4.3	0.4
SP02	150,000	5	Not Applicable - Created within mine pit				1.0
SP03	83,000	0.5	0.5	223,137	5.5	5.6	1.0
SP04	125,000	0.5	0.5	420,616	5.5	6.6	1.0

Notes:
(a) Required volume and pumping capacity to manage the 1:100-year flood event without overtopping. Does not consider pumping from the mine pits during the design event. Includes allowance for a 4-hour pump delay or interruption.
(b) Allowance for sediment accumulation.
(c) Allowance accounting for design, construction, and operation uncertainty.
(d) Including 1.0 m thick rock protection and filter layers.
(e) Including a 10 m wide band around the excavation perimeter

18.15.7 Pumping and Pipeline Systems

WSP (2024e) includes the feasibility study design for the following water pumping infrastructure:

- Seven sump pumping systems (with submersible pumps except for S03, which was designed to use a self-priming pump with the intake tethered to the pump cell location): S01 (same system as for S01b), S02, S03, S05, S06, S07, and S09.
- Five sedimentation pond pumping systems with a vertical turbine or low-lift pumps installed on floating barges or shore-mounted pumping wells: SP01, SP02A (from sedimentation pond SP02 to the environment – the Main Diversion Channel), SP02B (from SP02 to the Ore Processing Plant), SP03 and SP04.
- TSF Pond-to-Water Treatment Plan (WTP) pipeline: the design of the pump and barge at the TSF Pond was outside WSP's scope.
- Water Treatment Plant overflow pipeline to the Main Diversion Channel.

The WSP (2024c) technical report includes the design basis, the selected technologies, the design methods and results, and cost estimates, which account for the planned staged construction of the pumping systems. The construction staging was based on the mine development schedule.

The following paragraphs and Table 18-13 provide design details:

- All pipelines are designed for year-round operation. All pump barges or water intakes are equipped with agitation to mitigate surface freezing and allow pumps to operate all year.
- All pipelines will be profiled to facilitate free draining, maintaining a single high point with no intermittent low points. The exception is the TSF Pond-to-Water Treatment Plant pipeline, which will have low point drains and manually actuate in the winter months post-pump shut down due to constraints preventing the grading of intermittent low points. Pipelines encountering end-of-line slack flow necessitate a gooseneck vent to mitigate against pipe collapse and contribute to smooth pumping operation.
- Sedimentation pond pumps operate by hydrostatic level control while total suspended solids (TSS) limits are respected. The pumps are designed to activate with rising pond levels and deactivate when the pond level is low, or the TSS exceeds the specified discharge limit. The TSS limit adheres to Directive 019 with a maximum of 30 mg/L and a monthly average limit of 15 mg/L. In cases of pump shutdown due to high TSS, a predefined time delay precedes the automatic restart of the pump sequence, allowing particles time to settle. During commissioning, time values will be adjusted to ensure adequate settling time and prevent resuspension. Discharge into sedimentation ponds strategically occurs on the opposite bank of the sedimentation pond dewatering pumps to allow particles to settle out before environmental discharge.

Table 18-13: Pumping Systems Summary Characteristics

System	Pump Type	Duty	Standby	Pump location	E-house location	Total flowrate (m^3/h)	Pipeline Length (m) [TDH m]	Motor size per pump (hp)	Pipeline Nominal Size (HDPE 4710)
S01	Submersible	1	-	Barge	Shore	900	1615 [29]	200	16
S02	Submersible	2	-	Barge	Shore	1080	290 [52]	200	20
S03	Self Priming	1	-	Shore	Shore	1080	585 [69]	350	20
S05	Submersible	1	-	Barge	Shore	720	1140 [31]	200	16
S06	Submersible	1	-	Barge	Shore	720	515 [18]	100	16
S07	Submersible	1	-	Barge	Shore	720	205 [26]	200	10
S09	Submersible	1	-	Barge	Shore	1080	395 [29]	250	16
SP01	Vertical Turbine	1	-	Shore	Shore	1440	300 [13]	150	18
SP02A	Vertical Turbine	3	1	Barge	Barge	3600	2700 [113]	700	32 and 24 (a)
SP02B	Vertical Turbine	3	2	Barge	Barge	1351	3530 [134]	700	28 and 24 (a)
SP03	Lo-Lift	1	1	Shore	Shore	3600	10 [3]	250	28
SP04	Lo-Lift	1	1	Shore	Shore	3600	20 [8]	250	28
TSF to WTP	N/A	N/A	N/A	N/A	N/A	1200	3190 [75]	N/A	16

Notes: TDH : Total Dynamic Head. N/A: outside of WSP's scope. HDPE 4710: high density polyethylene
(a) SP02 pipelines require an initial ascent along the ramp to the in-pit SP02 sedimentation pond.

18.15.8 Water Treatment Plant

The Project WMP includes a water treatment plant (WTP) for the surplus water from the TSF Pond before its discharge to the environment. Based on Troilus's experience from the previous and current operations at the Site, the suspended solids in the TSF Pond are fine and passive gravity settling is not efficient. Instead, the WTP is used to remove the solids. Currently, Troilus is operating a 1,200 m^3 hour WTP at the location immediately near the TSF. The plant is approaching the end of its usable life cycle and may need to be replaced during the early years of the new planned operation.

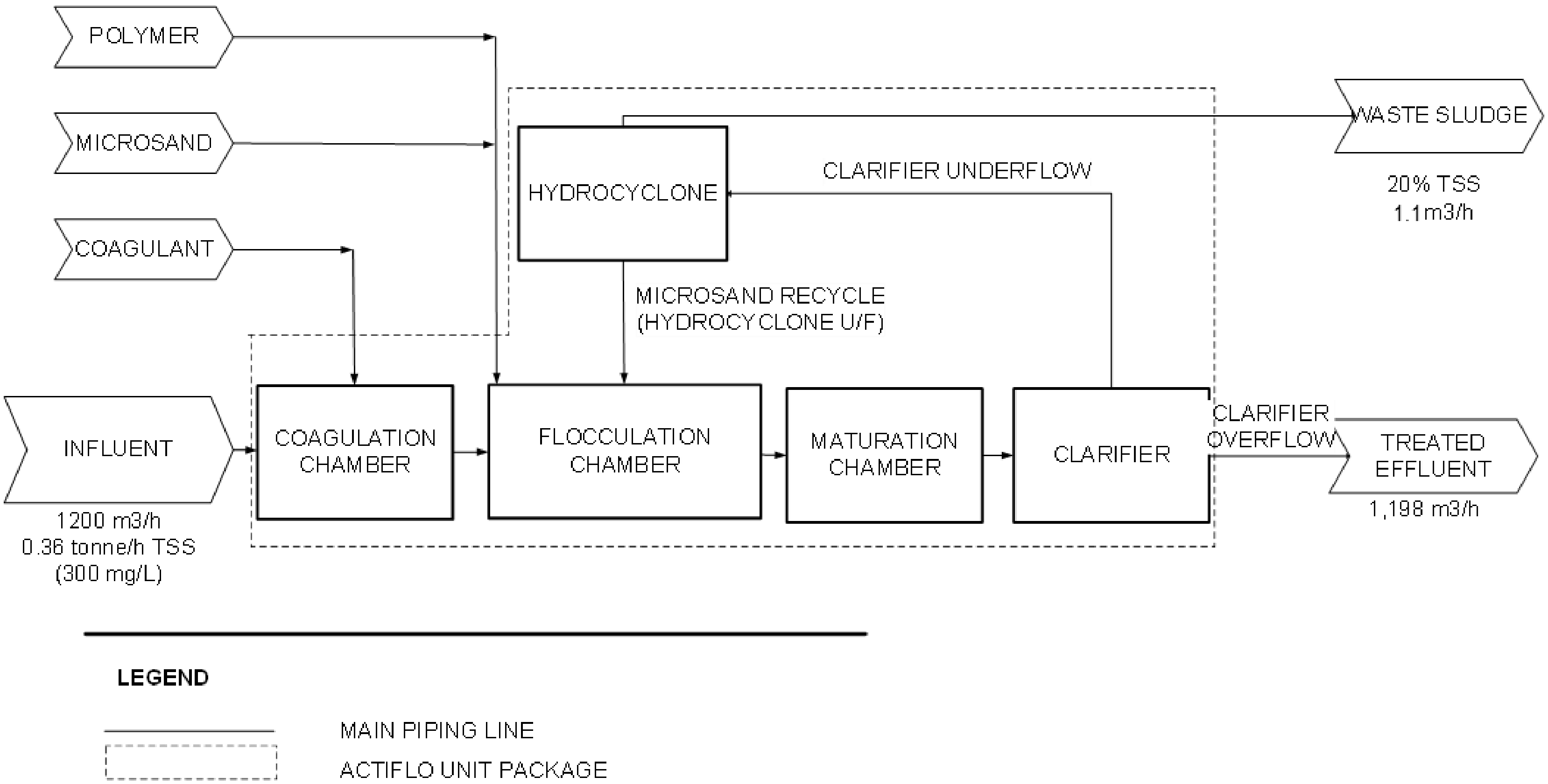
WSP (2024f) undertook a feasibility study design of the new WTP, including:

- A site selection study for the new plant. The current location would be covered by the TSF dam raises and by the waste dump placement.
- A water treatment technology comparison and selection analysis.
- A feasibility study design for the selected technology, including the list of major equipment, general arrangement drawings including major pipe routes, piping and instrumentation diagrams, preliminary instrument list, and estimation of capacity expenditures and preliminary operating expenditures.

Key points are provided in the next paragraphs:

- The new WTP was designed for a 1,200 m^3/hour treatment capacity, similar to the one in the current plant. The water balance analysis confirmed that this capacity is sufficient to control the TSF Pond water level under all climate scenarios.
- After analyzing several possible locations and consulting with Troilus, a location northwest of sedimentation pond SP01 was selected due to the reduced pipeline length from the TSF Pond.
- The treatment technology comparison concluded with the selection of a gravity separation, ballasted-flocculation lamella clarifier. The technology scored overall best in a multi-criteria comparison of treatment efficiency, reliability, complexity, maintenance requirements, and tolerance to changing influent TSS concentrations. Figure 18-9 summarises the block flow diagram for the selected technology.
- Treatment includes adding three chemicals: coagulant (ferric sulphate), flocculant (polymer), and ballast (microsand).
- An excavated 536 m^3 sludge pond is located near the plant, allowing for sludge storage during 21 days of plant operation at full capacity. From the sludge pond, the decanted solids would be trucked back to the TSF.

Figure 18-9 – Water Treatment Plant - Block Flow Diagram for the Selected Treatment Technology (Ballasted Flocculation System)



Source: WSP (2024)

18.15.9 Groundwater Management

WSP (2024d) compiled available hydrogeological data, including recent site investigations, and, also, documents the development of conceptual and numerical groundwater flow models covering the Project site. The model was used to estimate groundwater inflows to the four pits (87, J, Southwest, and X22), as well as the drawdown cone around the pits as they evolve during the lifetime of the project. The model was also used to verify the surface water-groundwater exchange around the DC1 Main Diversion Channel. The model results were used in the water management plan design and in the water balance analysis.

18.15.10 Site-Wide Water Balance

A site-wide water balance model was developed using GoldSim software to:

- confirm process water availability under various climate conditions
- estimate effluent flows from the site to the environment
- inform operational rules for the TSF Pond, the main process water source
- support the WMP design and provide the required information for permitting efforts and for the future project operation

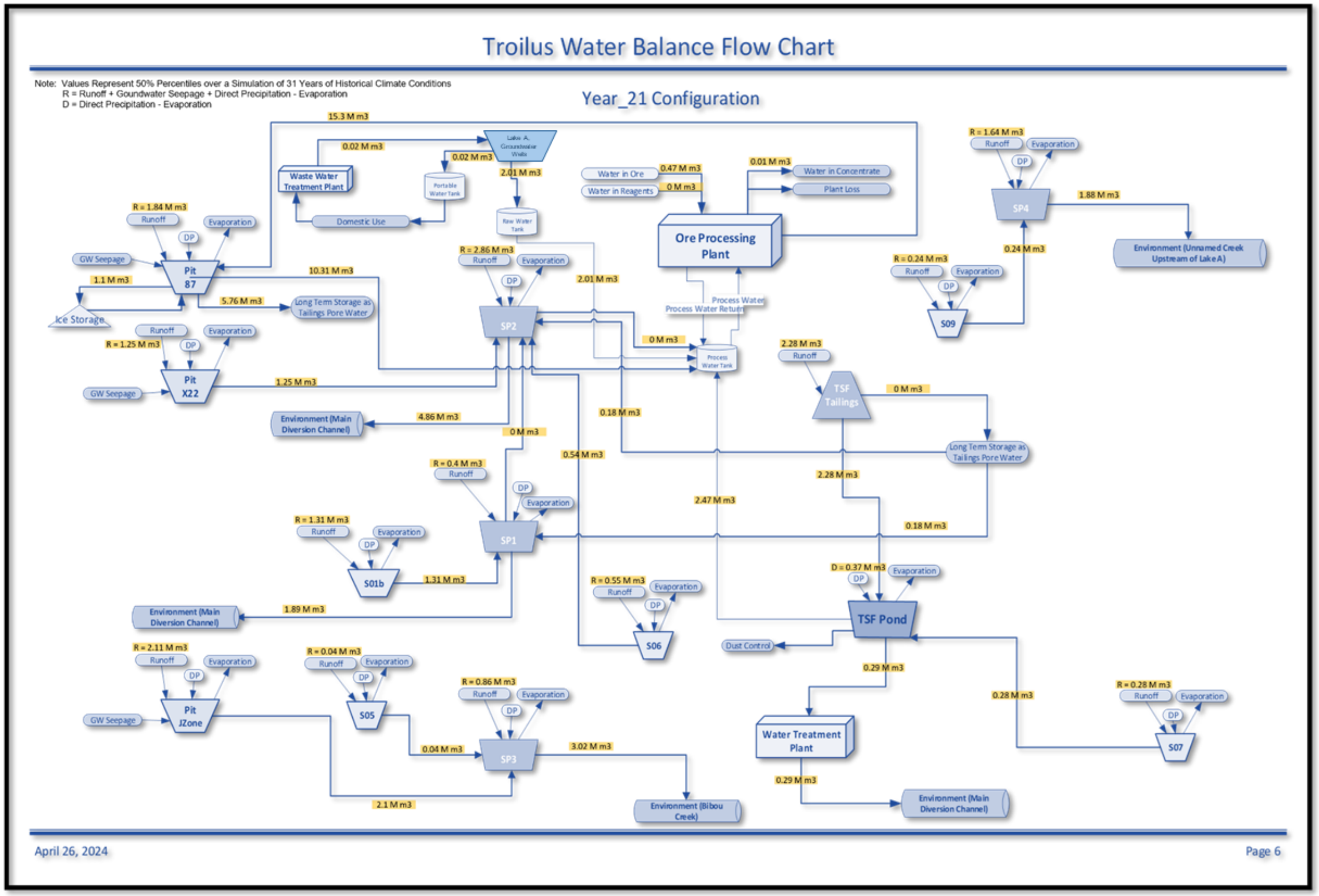
The water balance model simulated the relevant site water streams (site surface runoff, groundwater inflows to the pits, freshwater intakes to the plant, water transfer from the plant to the TSF and the reclaim from the TSF to the plan, and the contact water collection at the sedimentation ponds and their release to the environment). Figure 18-10 presents the developed flow logic diagram and the annual flows under historical average conditions and the final site development configuration. In addition, the water balance was verified under a wide range of climate conditions, both historical and considering climate change projections.

Key water balance results include the following:

- The contact water system can provide all the required process water supply even when accounting for seasonal effects and variable climate conditions.
- For all simulated configurations and climate scenarios, the TSF Pond water level was maintained between the minimum operating level (established to maintain the operability of the water reclaim system) and the maximum normal operating water level (established to maintain the environmental freeboard).
- The water treatment plant is required to discharge surplus water from the TSF Pond during wet climate conditions. The volume of water to treat varies depending on the site configuration and the climate conditions.
- The total effluent volumes discharged to the environment varies strongly with the site configurations and the climate scenarios. As examples:
    - For the end of Year 1 configuration and at full production capacity, the total annual effluent varies between 6.9 M m^3 and 22.0 M m^3. The average annual effluent volume is 11.9 M m^3 for the historical climate, and 12.9, 12.8, and 12.8 M m^3 for three climate change greenhouse gas emission scenarios, respectively.

- For the end of Year 9 configuration and at full production capacity, the total annual effluent varies between 7.9 M m^3 and 26.9 M m^3. The average annual effluent volume is 13.8 M m^3 for the historical climate, and 15.0, 14.8, and 14.8 M m^3 for three climate change greenhouse gas emission scenarios, respectively.

Figure 18-10: Site-Wide Water Balance Flow Logic Diagram and Annual Flows for Average Historical Climate Conditions and the Final Site Configuration



Source: (WSP, 2024)

18.15.11 Surface Water and Groundwater Monitoring

Surface water quantity and quality monitoring will be required as part of the project's environmental management program and to meet anticipated regulatory requirements.

A surface water monitoring system is proposed to meet these requirements during the construction, operation, and closure periods by extending the existing monitoring network. The proposed water monitoring system consists of various station types for which descriptions are provided in Table 18-14; additional surface water monitoring points will be required during the construction period as part of the construction care of water plan.

Table 18-14: Surface Water Monitoring Stations

Station Type	Monitoring Parameters	Regulatory Requirements	Monitoring Periods	Description of Monitoring Activities
Upstream water courses (Lake Amont and Lac B outlets)	WL, Q, and WQ	Directive 019 (MELCCFP, 2012)	CO and CL	Water level and discharge monitoring from all diversion channels use water level loggers (continuous) and established flow rating curves. WQ monitoring is continuous, composite, or grab in daily, weekly, monthly, bi-monthly, or tri-monthly intervals.
Sedimentation pond discharge (four sedimentation ponds and outlet from TSF Pond treatment plant)	WL, Q, and WQ	Directive 019 (MELCCFP, 2012)	CO	Water level and discharge monitoring for all sedimentation ponds use water level loggers (continuous) and established outflow rating curves. WQ monitoring is continuous, composite, or grab in daily, weekly, monthly, bi-monthly, or tri-monthly intervals.
Diversion channel discharges (main and secondary channels)	WL, Q, and WQ	Directive 019 (MELCCFP, 2012)	CO and CL	Water level and discharge monitoring from all diversion channels use water level loggers (continuous) and established flow rating curves. WQ monitoring is continuous, composite, or grab in daily, weekly, monthly, bi-monthly, or tri-monthly intervals.
Downstream water course (Lake A outlet)	WL, Q, and WQ	Provincial and federal environmental regulations	CO and CL	Water level and discharge monitoring at locations upstream and downstream of disturbance area uses water level loggers (continuous) and established flow rating curves. WQ samples are collected quarterly.

Notes: TSF = tailings storage facility; WL = water level; Q = discharge; WQ = water quality; CO = construction and operation; CL = closure.

WSP did not develop a groundwater monitoring plan as part of the current WMP:

- Troilus has already implemented a groundwater monitoring program, which considers the current infrastructure.

- The existing program will be updated during the next stages of the Project's development and, specifically, after the completion of the environmental impact analyses. WSP expects that the updating program will include groundwater monitoring wells downstream of the relevant developments (TSF, pits, dumps and stockpiles, and plant). The time frequency of data collection for groundwater level and, if applicable, groundwater quality is still to be decided.

18.16 Tailings Management Facility

The following paragraphs present high-level elements of the Tailings Management Feasibility Study (WSP, 2024g).

18.16.1 Existing Facility

There is one existing TMF at the Troilus mine site. The TMF commenced operations in 1996 and was raised annually from 1997 until 2009 when mining operations ceased, and the site entered the care and maintenance phase.

The initial design for the facility was carried out by SNC Lavalin Inc. (SLI). The original design report was not available; however, a 1997 report revision elaborated upon the key design assumptions. The SLI design concept was to construct a dyke initially to elevation 381 masl, subsequently to be raised to elevations 385 m and 390 m. Tailings were intended to be deposited on the upland areas to the northeast of the dyke and the supernatant water would rest against the embankment. Historical records indicate that the starter dyke for the main embankment of the facility was constructed up to nominal elevation 381 masl.

In general, the starter embankment, to elevation 381 masl, was designed as a till embankment with sand and gravel toe drain. Upstream slopes of 2H:1V, downstream slopes of 3H:1V, and an 8 m wide crest were proposed. Rock riprap was placed on the upstream face and a thin granular travelling layer was placed on the crest.

The concept for the development of the TMF was revised by Golder in 1997. The concept selected was to raise the tailings facility by constructing a series of upstream raises 2 to 2.5 m high using tailings placed on top of the tailings beach. A filter zone was constructed to allow the passage of exfiltrating water, and mine waste was used for erosion protection. The TMF was raised incrementally using this concept, except that the annual raises were typically 1.5 m high.

Whereas the raise construction methodology was successful, issues were experienced with high levels of suspended solids in the TMF water pond, which posed water management challenges. Several iterative adjustments to the dyke raise heights and pond size were considered during the operational phase to manage these challenges.

In 2004, a study was undertaken to assess the potential to construct a toe berm along the main embankment using mine waste rock. The study demonstrated the beneficial effect of the toe berm, and the berm was constructed in 2005. The berm crest elevation varied from 394 to 397 m, which corresponded to the crest of the 2005 raise.

In 2006, the South Dyke was constructed up to a crest elevation of 402 masl. This dyke is a till core, rock shell dam with a sand and gravel filter between the till core and the downstream rock shell.

Upstream and downstream slopes for this structure are 2H:1V and 2.5H:1V, respectively. The dyke was not raised subsequent to its original construction.

Subsequently, the facility was raised annually until 2009. The last raise crest elevation varied between 400 and 402 masl. In 2010, the mine received a certificate of authorization for a closure plan of the facility.

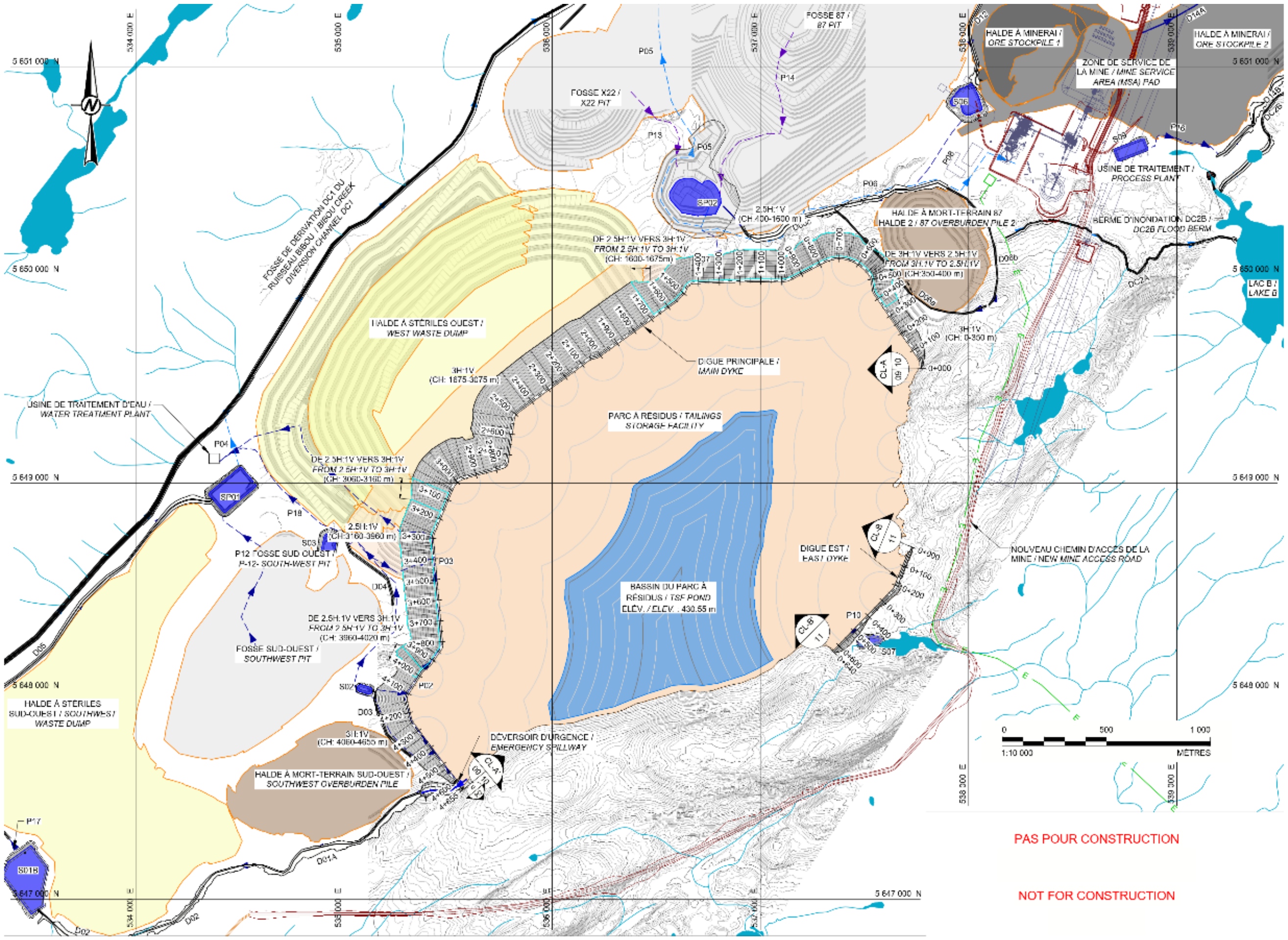
18.16.2 Feasibility Design of Tailings Storage Facility Expansion

The FS design of the TMF stores 169 Mt. The design assumes a raise of the existing dam (Main dyke) to a final elevation of 435 masl and construction of a saddle dyke (East dyke) along the low point of the mountain slope on the eastern side of the TMF, also to 435 masl (Figure 18-11).

The feasibility design of the Main dyke raises, and East dyke is for downstream construction. For the Main dyke in particular, measures have been incorporated into the design to migrate from upstream to downstream construction, resulting in improved geotechnical stability. The downstream construction method provides additional storage capacity and a lower environmental risk than the upstream and centreline construction adopted for the existing TMF.

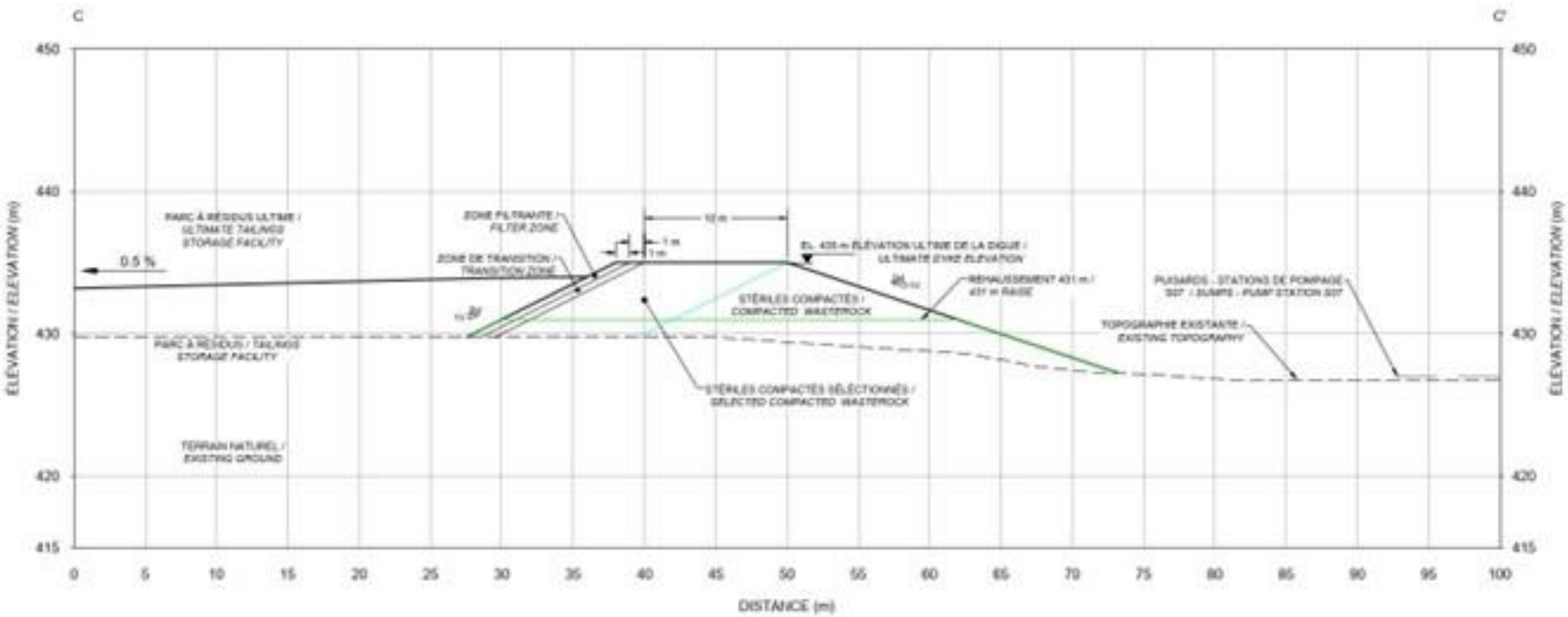
The final configuration of the TMF is shown in the plan view in Figure 18-11. A cross-section of the Main dyke is shown in Figure 18-12 and that of the East Dyke is shown in Figure 18-13. The dams will be raised to 435 masl.

Figure 18-11: Plan View of Tailings Storage Facility at Final Configuration



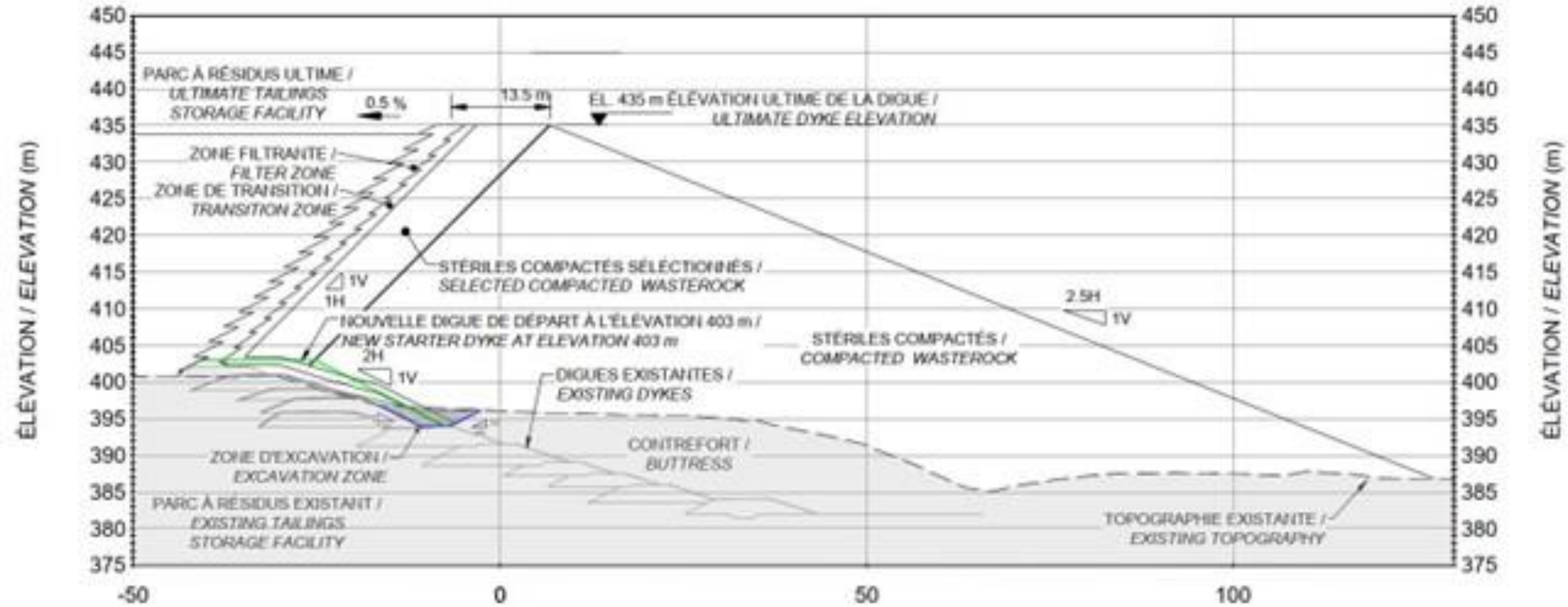
Source: WSP (2024)

Figure 18-12: Cross-Section of Main dyke w/ New Starter Dyke Shell in green (historic dyke raises simplified for clarity of presentation)



Source: WSP (2024)

Figure 18-13: Cross-Section of East Dyke



Source: WSP (2024)

The feasibility design was developed using extreme loading conditions per the Global Industry Standard for Tailings Management (GTR 2020) and the criteria of Directive 019, which outlines applicable controls for the design and the construction and operation of TMFs in Québec. Canadian Dam Association (CDA) dam safety guidelines are used broadly and depend on a risk-based classification. Consequence classification is determined for a TMF based on incremental losses that a failure of the TSF may inflict on downstream or upstream areas. As the earthquake and flood criteria will be consistent with GTR (2020), which presumes a 1:10,000-year earthquake and flood criteria as defined in CDA (2013), the requirements of the CDA (2013) and consequence classification will not be defined for this study.

Stability analyses of the dams were completed using SLOPE/W, a commercially available limit equilibrium analysis software package using the Morgenstern-Price method of analysis, which concurrently satisfies both force and moment equilibrium. The FS embankment designs meet the geotechnical stability requirements under static and extreme seismic loading conditions.

A stress-deformation analysis was carried out to calculate the stress distribution, horizontal and vertical displacements, as well as the pore water pressure regime. The stress-deformation analysis also allowed to assess the deformation around the transition and filter layers due to the construction and tailings deposition. PLAXIS finite-element model (FEM) software was used to simulate a time-dependent fully coupled flow-deformation analysis of construction of the dyke and the deposition and consolidation of tailings. The model permitted to estimate the pore water pressure build-up and dissipation, as well as the subsequent settlement through the sequences of staged construction. The analysis indicates that the existing embankment filters will experience minor deformation and should perform as per their design intent. The simplified dynamic displacement analyses were carried out to assess the performance of the main dyke during the design earthquake after the Bray and Macedo (2019) method. The results indicate that the displacements of the main dyke induced by the design earthquake remain negligible.

18.16.3 Tailings Deposition Plan

The thickened tailings (approximately 54.3% wt w/w solids) will be produced at the processing plant and pumped for permanent storage. The TMF will reach its full capacity during Year 10 and tailings are planned to be disposed into mined out pits until the end of Mine Life (Year 22). In-pit tailings disposal will progress sequentially into the SW pit, pit J4 and then pit 87. The mine plan was developed with this constraint of concept for migration to in-pit disposal in the later years of mine development (AGP, 2024).

The storage capacities per location and on a cumulative basis are presented in Table 18-15.

Table 18-15: Tailings Management Capacity

Tailings Disposal Location	Volume (Mm^3)	Dry Tailings Mass* (Mt)	Cumulative (Mm^3)	Cumulative (Mt)
TMF (dyke elevation 435 masl)	113	169	113	169
Pit SW (elevation 367masl)	62	93	175	262
Pit J4 (elevation 169 masl to saddle with pit 87)	26	39	201	301
Pit 87 (elevation 98 masl)	52	79	253	380
Total tailings storage	**253**	**380**	**253**	**380**

*Tailings dry density of 1.5 t/m^3

The tailings storage capacity is sufficient for the LOM resource but with a temporary shortfall occurs within the current mine plan. Indeed, the pit development, tailings production and tailings storage capacity indicate a shortage of approximately 15 months (22 Mt) around Year 17. The TMF, SW pit and J4 pit reach their full capacity before pit 87 is ready (mined out) for in-pit tailings disposal. There are a few solutions that could be implemented to address this shortfall, and these be advanced in future design stages to validate their feasibility:

- The SW pit capacity could be increased by building a ring dyke around it. The surrounding waste rock dump and TMF could contribute to the stability of the retaining structures by acting as stability berm. This concept is consistent with the closure concept of covering the SW pit completely with waste rock.
- The TMF capacity could be increased with additional upstream raises above the dyke elevation of 435 masl. An increase of approximately 5 m (to the elevation of 440 masl) could be sufficient to accommodate the 22 Mt. Upstream raises have been used at the site historically, such that this approach can be considered feasible.
- The tailings deposition planning, tailings consolidation and formation of persistent ice lenses are factors impacting the storage capacity. The tailings management practice should be implemented to increase tailings storage.

Tailings Management Facility

A deposition plan for the TMF was completed to optimize the tailings storage, determine construction staging, develop construction material quantities and to inform the spillway design and water management for the TMF. The concept of an internal pond away from the perimeter and draining perimeter embankments is conserved. This concept results in lower pore pressures in the tailings adjacent to the perimeter dykes and has a positive effect on stability. Deposition modelling was completed using commercially available Muk3D software in 6 months increments.

The objectives of the deposition plan are:

- form an operational dry beach of at least 250 m against the upstream slope of both the Main Dyke and East Dyke
- achieve suitable water clarity by having a minimum decant pond depth of 3 m at the position of the pumping station

- achieve a decant pond location suitable for the plant's process requirements
- reduce movement of the decant pump barge
- achieve the conceptual level closure design

In-pit Deposition Concept

After the TMF will be full, the tailings will be disposed in the mined-out pits. The pits' capacity to store tailings is limited by geological constraints. At this conceptual design stage, the maximum tailings elevation is limited by the bedrock surface elevation along the pit rim. This approach simplifies the ground stability considerations by limiting the potential interaction of tailings deposition and the surrounding overburden slopes. Since the current groundwater elevation is above the bedrock surface, the above limitation will allow the disposal concept to operate with groundwater flow into the facility, creating a hydraulic trap and mitigating water seepage from the tailings body toward surrounding groundwater.

The tailings in the SW pit will be placed up to the lowest bedrock elevation (367 masl) along the pit rim. The J4 pit and 87 pit will be connected by a saddle between the pits (at the elevation of 169 masl); therefore, pit J4's storage capacity will be limited to this elevation. Pit 87 will only be filled up to elevation 98 masl during the last years of the mine life but will have a greater storage capacity if needed.

The tailings will be deposited by 2 to 4 spigots from one side of the pit and return water ponds against the opposite site of the pit. A barge will be installed to pump reclaim water back to the mill. A tailings delivery pipeline will need to be constructed to deliver tailings to a few discharge points with limited lateral displacement. The discharge points will be raised, typically along the ramp as the tailings level rises.

The pit water level will be controlled by pumping from a floating barge in the pit. This system can continue to operate from operation to closure and allow efficient transition to pit lake closure operation for the J4 and 87 pits. The geochemistry of tailings is assumed to be favourable but if any issues are identified, the proposed water management permits to create an inward groundwater gradient during operations and mitigate potential groundwater impacts by avoiding potential exfiltration toward groundwater.

18.16.4 TMF Construction Sequence

The design assumes a downstream raise of the existing dyke (Main Dyke) to a final elevation of 435 masl and construction of a saddle dyke (East dyke) along the low point of the mountain slope on the eastern side of the TMF, also to 435 masl. The East Dyke is required once the tailings reach an elevation of 428 masl within the TMF, as they can no longer be naturally contained within the TMF sub-catchment of the mountain. It is noted that downstream raises of the facility are constrained by the location and ultimate footprint of the pits adjacent to the toes of the facility. The ultimate configuration of the pits constrains the ultimate elevation and the capacity of the TMF under the selected development methodology.

For the Main dyke, measures have been incorporated into the design to migrate from the existing upstream raised facility to downstream construction, resulting in improved geotechnical stability.

A new starter dyke will be constructed up to the elevation of 403 m. Based on an annual tonnage of 18.18 Mt, the raise will be constructed in stages of approximately 4 m/yr until Year 8 and 2 m/yr for Year 9 and Year 10. The final elevation will be 435 m. The upstream slope of the future raises will be globally 1.0H:1V, and the ultimate downstream slope of the dyke will vary between 2.5H:1V and 3.0H:1V, depending on the sector.

The western section of the TMF perimeter embankment will be constructed contiguous with the proposed waste rock pile. An integrated waste rock/tailings construction approach will be implemented to promote the robustness of the tailings containment structure and the long-term performance of the tailings storage facility. It is noted that the waste rock pile will be substantially higher than the TSF embankment.

18.16.5 TMF Monitoring Program

A series of instruments is proposed in order to confirm the geotechnical performance of the structures associated with the TMF:

- six to eight inclinometers located in the embankments to monitor lateral movements
- 10 to 12 vibrating wire piezometers in the embankments and the tailings to monitor pore pressures and pore pressure variations
- 15 to 20 survey markers on the embankment to monitor vertical deformations
- eight to ten thermistors located in the embankments in the tailings to measure the temperature profile
- a series of monitoring wells, to be determined, to monitor seepage water quality

The monitoring program, including trigger alarm response plans (TARPs) will be augmented by a program of regular visual inspection, and will be outlined in a detailed operation, maintenance, and surveillance (OMS) manual which will be prepared prior to commissioning of the facility.

18.16.6 TMF Water Management

Drainage from the tailings beach and mountain slope will be collected by gravity to the TMF pond. The drainage includes both natural runoff and water exfiltrating from the deposited tailings. Pond water will be pumped for recirculation to the mill and, if there is surplus, for discharge to the environment after treatment for total suspended solids (TSS) removal via the existing treatment plant for the first 2 years and after by its eventual replacement. TMF water management will occur in much the same way as historical operations. The TMF should operate with at least 250 m long upstream tailings beach to control seepage rate and a 2 m pond freeboard, both of which are achievable based on geometric modelling carried out for the study.

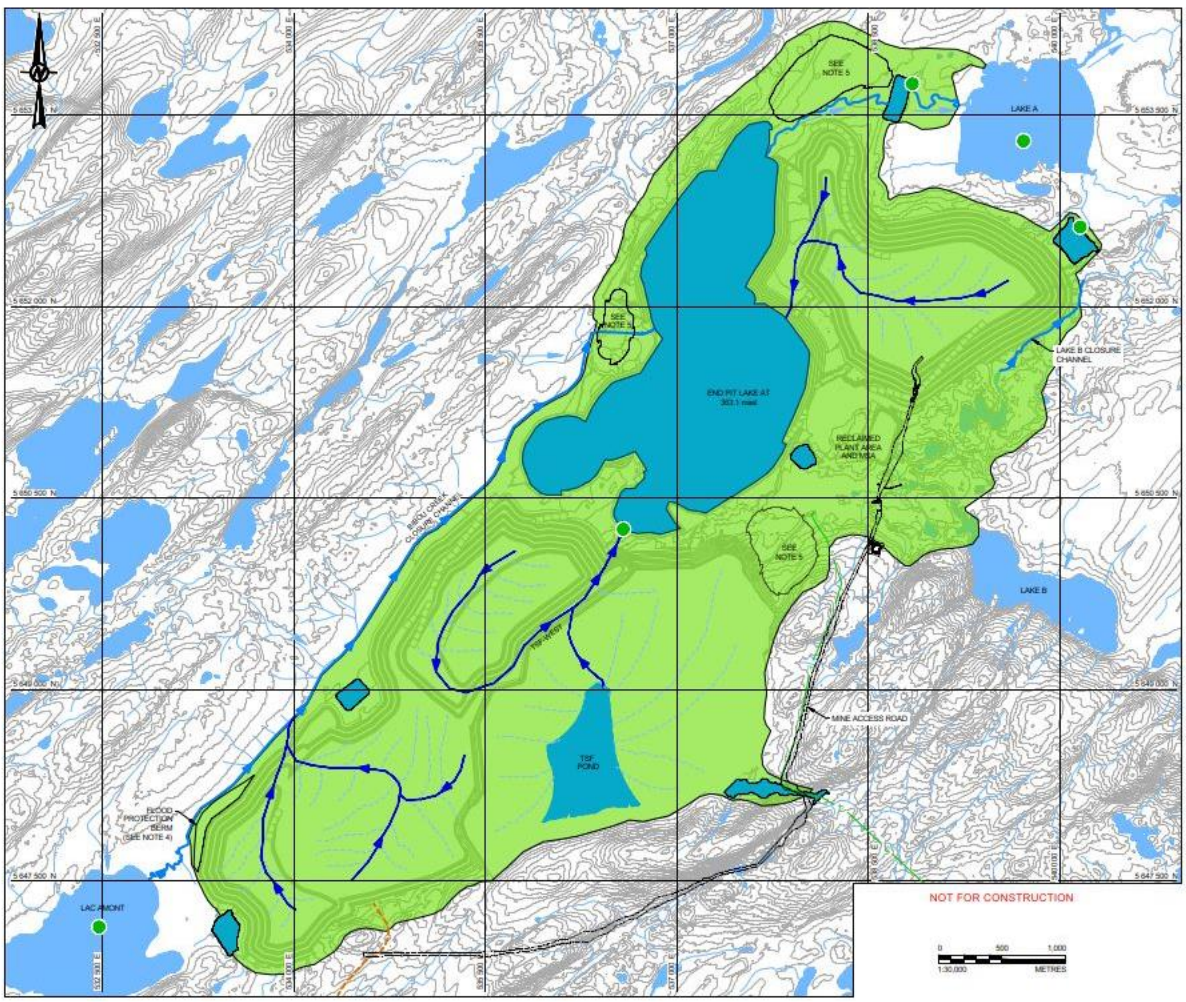
The pond and the beach geometry allow containing the regulatory Directive 019 environmental project flood. The emergency spillway is designed to manage the probable maximum flood.

18.16.7 Closure Considerations

The detailed closure design of the TMF will be developed in the future, as the geometry and final volume of the facility is likely to evolve during mining. The current concept for TMF closure is expected to resemble the concept which was approved for closure of the existing facility and will be integrated into the overall mine closure strategy. Figure 18-14 includes a conceptual closure drainage figure. Also:

- At the end of operations, a closure spillway channel would be built to connect the TMF Pond to the flooded Pit 87 Lake located north of the TMF. The channel would allow lowering the pond water level to increase the long-term freeboard relative to the embankment crest.
- The TMF surface would be reprofiled to allow efficient drainage towards the TMF Pond or the new spillway channel.
- Granular material would be placed on the TMF Pond bottom to avoid the wind resuspension of the tailing fines. The tailings beach would be vegetated to create a closure landscape and to mitigate beach erosion.
- Toe ditches and sumps would be reconfigured and integrated into the overall closure drainage plan.

Figure 18-14: Conceptual Closure Plan



Source: WSP (2024)

19 MARKET STUDIES AND CONTRACTS

A concentrate marketing study (Ocean Partners Study) was completed by Ocean Partners Limited UK (Ocean Partners) dated 4 June 2024.

19.1 Summary

This section examines potential smelting and refining terms for the copper concentrate and the gold in the concentrate expected to be generated from the Project. As well, doré gold will be produced from gravity separation in the process plant which will be sold into a different market.

Troilus Gold retained Ocean Partners to perform a review of the Concentrate Market for the future production of concentrate from the Project. The Ocean Partners Study has been reviewed by the QP and the results support the assumptions in this Report, and the following section is [reproduced/derived] from the Ocean Partners Study.

Based on the specifications provided, the Troilus Cu-Au flotation concentrate would be regarded as a very clean material with a slightly lower than average copper grade but a significant gold content. It can be assumed with some confidence that a market will readily be found for the proposed 70kdmt/a of production.

Glencore's Horne smelter may be a 'natural' home for a significant proportion of the Troilus flotation concentrate. It is understood that Troilus Gold is in direct contact with Glencore in this regard. This report is focused on potential alternative off-take arrangements.

The range of commercial terms for four potential options for selling the Troilus concentrate in the international market are detailed in this report. Potential terms for the material described containing 14.9% Cu, 111 g/t Au and 303 g/t Ag are summarised in Table 19-1 below:

Table 19-1: Commercial terms (LOM average)

	Direct sales to Aurubis	Direct sales to Boliden	Direct sales to PPC (Japan)	Sales to Ocean Partners
Delivery	CIF FO Brunsbuttel, Germany	CIF FO Skelleftea, Sweden	CIF FO Saganoseki, Japan	DAP Montreal warehouse.
Inland truck freight	$72.5/wmt	$72.5/wmt	$72.5/wmt	$72.5/wmt
Port handling costs	$20/wmt	$20/wmt	$20/wmt	Quarterly freight credit to be agreed equivalent to CIF FO Taiwan basis 5,000 wmt shipments for Seller's account. ($110/wmt based on assumed freight rates to Asia).
Ocean Freight	$65/wmt	$80/wmt	$90/wmt	
Freight adjustment	-	-	-	
Treatment charge	Benchmark + 10%. Assume $77/dmt	$85/dmt	Benchmark. Assume LT Average $70/dmt	Benchmark -10% Assume LT Average $63/dmt
Copper payment	96.6% MD 1.1 units	96.6% MD 1.1 units	96.5% MD 1.1 units	96.6% MD 1.1 units
Copper refining charge	$0.077/lb	$0.85/lb	$0.07.lb	$0.063/lb.
Gold payment	98% MD 2g/t	97% MD 2g/t	98.15%	97.5% MD 2g/t.
Gold refining charge	$5.0/oz	$5.0/oz	$4.0/oz	$5.0/oz
Silver payment	97% MD 20g/t	97% MD 20g/t	90%	90%
Silver Refining charge	$0.50/oz	$0.50/oz	$0.40/oz	$0.50/oz
Penalties	Assumed to be a clean concentrate attracting no penalties.			
Estimated Ocean Transit Time	30	45	75	75
QP	2 MAMA	2 MAMA	2 MAMA	M+1 or 2 MAMA with option to price at any point before onset of QP.
Payment	30 days after arrival	10 days after arrival	30 days after arrival	End of Month after month of delivery to warehouse. Earlier financing available at 3M LIBOR + 5.5%.

The treatment and copper refining charges and freight costs presented above are long-term averages intended for financial modelling purposes. As described in the discussion of the global copper concentrate market included in this report, TCRCs are likely to be declining to levels below long-term averages in the years immediately following the proposed project start date in the second half of this decade.

The benchmark-based terms proposed can be regarded as conservative assumptions reflecting the desire of financing banks to see calculations based on direct sales to smelters under long- term contracts. It is probable that at least a portion of the project's output could be sold to traders under either long-term or spot contracts. Currently, Ocean Partners is willing to offer purchase terms at Benchmark -10%.

It will be important for Troilus to ensure that the overall commercial terms attached to sales remain competitive throughout the entire life of the project. This report proposes several marketing strategies intended to mitigate this risk.

It is likely that international smelters will require minimum lot sizes of at least 5kdmt in bulk. If Troilus Gold does opt for any of the available options for direct smelter sales, then Ocean Partners could step in between the contract and finance from holding certificate to contractual payment terms at 3M LIBOR +5.5%. Ocean Partners can also assist with pricing services before the onset of the direct smelter quotational period (QP).

19.2 Introduction

Troilus is evaluating a project to re-start the former Troilus mine located northeast of the Val-d'Or district of Québec, Canada. Mining at the site, which was formerly owned by Inmet Mining Corporation, ended in April 2009 and the process plant at the site stopped operation in June 2010.

It is intended that a new process plant at the mine site will produce a gravity gold concentrate together with approximately 70kdmt/a of a copper/gold flotation concentrate. The flotation concentrate will be sold to third-party smelters for recovery of the contained copper, gold, and silver.

The specification of a copper concentrate produced from metallurgical testwork on ore from an area of the mine known as J-Zone is summarised in the Table 19-2 below:

Table 19-2: Copper Concentrate Specification for J-Zone from Metallurgical Testwork

Element		Unit	Value	Element		Unit	Value
Silver	Ag	g/t	303	Manganese	Mn	ppm	89
Arsenic	As	ppm	86	Molybdenum	Mo	ppm	1680
Gold	Au	g/t	111	Nickel	Ni	ppm	600
Barium	Ba	ppm	31	Lead	Pb	ppm	719
Bismuth	Bi	ppm	96	Palladium	Pd	g/t	0.07
Calcium	Ca	%	2.0	Platinum	Pt	g/t	0.07
Cadmium	Cd	ppm	64	Sulphur	S	%	38.3
Cobalt	Co	ppm	462	Antimony	Sb	ppm	13
Chrome	Cr	ppm	220	Selenium	Se	ppm	45
Copper	Cu	%	14.9	Silicon	Si	%	1.49
Fluorine	F	%	0.01	Strontium	Sr	ppm	103
Iron	Fe	%	31.5	Uranium	U	ppm	1.4
Mercury	Hg	ppm	3	Zinc	Zn	ppm	9120
Magnesium	Mg	%	1.38				

It is understood that Troilus Gold is in direct contact with Glencore regarding supplying concentrate to the Horne smelter which is also located in Québec.

The purpose of this report is to provide a review of alternative options for marketing the Troilus flotation concentrate including: -

- current market demand with comments on specific aspects impacting marketability
- 5 to 10-year outlook on market demand with comments on specific aspects impacting marketability
- detailed current commercial terms available and locations of potential buyers, and 5 to 10-year outlook on commercial terms available and locations of potential buyers
- inland and ocean freight analysis

19.3 Copper Concentrate Quality

19.3.1 Copper Concentrate Quality – General Overview

The quality of an individual copper concentrate is usually defined by its copper grade, precious metal content and the presence of otherwise of any deleterious elements.

19.3.2 Copper Grade

The most common copper bearing mineral recovered in sulphide concentrates is Chalcopyrite ($CuFeS_2$) which contains 35% copper. Higher grade sulphide concentrates may contain some Bornite (Cu_5FeS_4) which contains 63% copper. Incomplete liberation of the copper bearing minerals at the grinding stage means that the overall copper grade of the final concentrate will be diluted by the presence of gangue minerals which are usually present in the form of oxides of silicon, calcium, aluminium, and magnesium.

For concentrates containing below 28.57% Cu and above 20% Cu the proportion of copper in concentrate is usually calculated by a 'one-unit deduction'. Hence the proportion of copper paid for in a lower grade concentrate is less than would be received for higher grade material. For example: -

- for a concentrate containing 25% copper (25% - 1%)/25% = 96.0% payable
- for a concentrate containing 28.5% copper (28.5% - 1%)/28.5% = 96.49% payable

For concentrates containing below 20% Cu and above 10% Cu the proportion of copper in concentrate is usually calculated by a '1.1-unit deduction'. For grades below 10% the deduction can be anywhere from 1.2 units to 1.5 units.

The net value of a copper concentrate is derived by deducting treatment and refining charges together with any penalties for deleterious elements from the value of the paid metal. Since the treatment charge for a copper concentrate is expressed in $/dmt of concentrate it is proportionally higher relative to the copper content for lower grade concentrates.

19.3.3 Precious Metals

Precious metals contained in copper concentrates are recovered at the electro-refining stage in the form of an insoluble 'anode sludge.' Most copper concentrate purchase contracts will include terms for the payment of contained silver and gold above certain minimum thresholds.

As noted elsewhere in this report, Chinese smelters have come to dominate the market for custom copper concentrates in recent years. However, it should be noted that many Chinese copper smelters have not yet established a consistent supply of gold bearing copper concentrates. As a consequence, they have not had the opportunity to optimise their operations for the recovery of gold and the payment scales they can offer are lower than can be found elsewhere. It should also be noted that TCRCs offered by Chinese smelters for material with a significant silver content will reflect a non-refundable, 13% VAT attached to the silver content in copper concentrates imports.

Smelters elsewhere in Asia and in Europe are therefore currently the preferred destinations for copper concentrates with a significant precious metal content. Table 19-3 displays typical payment terms for precious metals in copper concentrates.

Table 19-3: Typical Payment Terms for Precious Metals Contained in Copper Concentrates

<table>
<tr><th colspan="2"></th><th>China</th><th>Japan / Korea</th><th>Europe</th></tr>
<tr><td rowspan="7">Gold</td><td>< 1 g/t</td><td>None</td><td>None</td><td rowspan="7">Pay 97 to 98% minimum deduction 0.7 to 1.0 g/t</td></tr>
<tr><td>1 – 3 g/t</td><td>90%</td><td>90%</td></tr>
<tr><td>3 – 5 g/t</td><td>92%</td><td>94%</td></tr>
<tr><td>5 –10 g/t</td><td>93% to 94%</td><td>96%</td></tr>
<tr><td>10 – 15 g/t</td><td>95 to 96%</td><td>96.5 – 97%</td></tr>
<tr><td>15-50 g/t</td><td>96 – 97%</td><td>97.5%</td></tr>
<tr><td>> 50 g/t</td><td>97-97.5%</td><td>98.00% - 98.20%</td></tr>
<tr><td></td><td>RC</td><td>$5.00/oz.</td><td>$6.00/oz</td><td>$5.00/oz</td></tr>
<tr><td>Silver</td><td>Pay</td><td>Pay 90% if the content exceeds 30g/t.</td><td>Pay 90% if the content exceeds 30g/t.</td><td>Pay 97% minimum deduction 20-30g/t</td></tr>
<tr><td></td><td>RC</td><td>$0.40/oz</td><td>$0.50/oz</td><td>$0.30/oz</td></tr>
</table>

19.3.4 Chinese Import Limits

In 2006 China introduced new legislation named 'Harmful Content of China Imported Copper Concentrates (No. 49/2006)' under the China Import and Export Inspection Act. This imposed import limits on certain deleterious elements contained in copper concentrates. As a consequence, Chinese port authorities became empowered to reject copper concentrate imports containing in excess of 0.5% arsenic, 6% lead, 1000 ppm fluorine, 100 ppm mercury or 500 ppm cadmium.

Certain complex materials with a significant gold content are able to access the Chinese market by obtaining classification as a gold concentrate either directly or as part of a blend. Until recently, to achieve this the gold content of the imported material needed to exceed 20g/t and its value had to exceed that of the contained copper. In addition, the concentrate needed to contain <20% moisture and have a size analysis of at least 50% -200# (74µm). If these conditions were met there was no limit for the arsenic content. However, these regulations will change during the second half of 2021. It is understood that under the new conditions, material containing >15g/t Au will be regarded as a gold concentrate however the maximum permissible arsenic content will be 3.5% As between 15g/t and 60 g/t Au and 6.5% As above 60 g/t Au.

19.3.5 Deleterious Elements

Smelters will charge penalties for treating concentrates in which the level of certain deleterious elements exceeds a given threshold. These penalties reflect the additional costs associated with processing and disposing of such materials together with any additional environmental controls that may be required. Typical penalty limits and deductions are summarised in Table 19-4 below:

Table 19-4: Typical Penalties for Deleterious Elements in Copper Concentrates

		Comment	Typical Limit	Typical penalty deduction per tonne of concentrate
Arsenic	As	Reduces conductivity of refined copper. Due to toxicity, the smelter incurs extra costs associated with control of off-gases and dust & slag disposal.	0.20%	\$2 to \$3 per 0.1% up to 0.5% \$5 to \$8 per 0.1% up to 1% \$8 to \$12 per 0.1% above 1%
Alumina	Al_2O_3	Increases melting point and viscosity of slag. Smelter incurs extra heating costs	3%	\$1 to \$2 per 1%.
Antimony	Sb	Reduces conductivity, annealability and drawability of refined copper. Toxicity issues	0.05%	\$1 to \$2 per 0.01%.
Bismuth	Bi	Low concentrations cause rod cracking and poor drawability	0.02%	\$1.5 to \$3.0 per 0.01%
Cadmium	Cd	Due to toxicity. The smelter incurs extra costs associated with control of off-gases and dust & slag disposal.	0.03%	\$1.5 to \$5.0 per 0.01%
Chlorine	Cl	Forms corrosive hydrochloric acid	300ppm	\$1 to \$3 per 100ppm
Fluorine	F	Forms corrosive hydrofluoric acid	300ppm	\$1 to \$2 per 100ppm
Lead	Pb	Toxicity issues. Anode passivity issues in electro- refining.	1%	\$1 to \$5 per 1%
Magnesia	MgO	Similar to alumina.		
Mercury	Hg	Toxicity issues. Can contaminate smelter acid.	5ppm	\$0.1 to \$5 per 1ppm
Molybdenum	Mo	Scaling issues in furnace.		
Nickel + Cobalt	Ni + Co	Limits on Ni content in LME grade A cathode. Removed from refinery electrolyte as nickel sulphate. Cobalt follows nickel at electrolyte purification stage.	0.50%	\$1 per 0.1%
Selenium	Se	Makes copper cathode prone to cracking. Can cause copper losses to anode slimes.	300ppm	\$1.50 per 100ppm
Silica	SiO_2	Similar to alumina, but less of a problem since silica is added to furnace as a flux. Excessive silica content will lead to miner incurring extra cost through TC and freight costs.	10%	\$1 per 1%
Tellurium	Te	Similar to Selenium		
Uranium	U_3O_8	Smelters can be reluctant to accept material with an elevated uranium content because radioactive decay products accumulate in smelter flue dust. Limit for acceptance in China is 1 B q/g (approx. 80ppm) or 10 x background count.		
Zinc	Zn	Reports to slag increasing viscosity and potentially leading to copper losses.	3%	\$1 to \$5 per 1%

19.3.6 Troilus Gold Project – Concentrate Quality

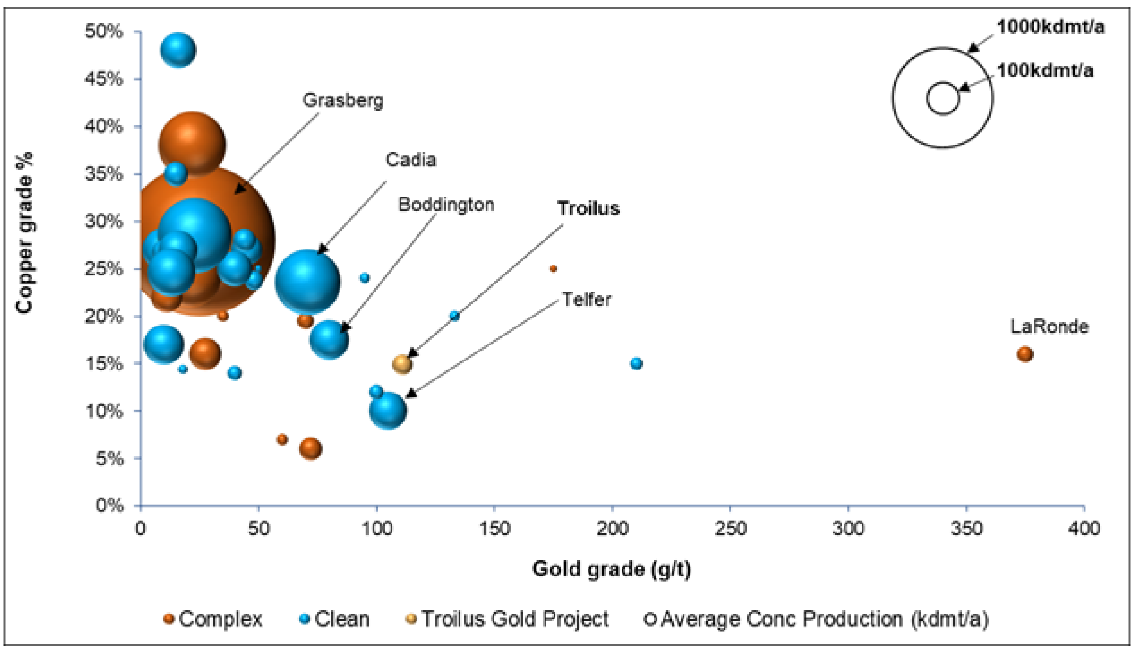
The specification of a copper concentrate produced from metallurgical testwork on ore from an area of the mine known as J-Zone is summarised in Table 19-5 below.

Table 19-5: Copper Concentrate Specification for J-Zone from Metallurgical Testwork

Element		Unit	Value	Element		Unit	Value
Silver	Ag	g/t	303	Manganese	Mn	ppm	89
Arsenic	As	ppm	86	Molybdenum	Mo	ppm	1680
Gold	Au	g/t	111	Nickel	Ni	ppm	600
Barium	Ba	ppm	31	Lead	Pb	ppm	719
Bismuth	Bi	ppm	96	Palladium	Pd	g/t	0.07
Calcium	Ca	%	2.0	Platinum	Pt	g/t	0.07
Cadmium	Cd	ppm	64	Sulphur	S	%	38.3
Cobalt	Co	ppm	462	Antimony	Sb	ppm	13
Chrome	Cr	ppm	220	Selenium	Se	ppm	45
Copper	Cu	%	14.9	Silicon	Si	%	1.49
Fluorine	F	%	0.01	Strontium	Sr	ppm	103
Iron	Fe	%	31.5	Uranium	U	ppm	1.4
Mercury	Hg	ppm	3	Zinc	Zn	ppm	9120
Magnesium	Mg	%	1.38				

The Cu and Au grades of the proposed Troilus flotation concentrate are compared with a peer group of 40 internationally traded copper-gold concentrates (>10 g/t Au & 5% Cu) in Figure 19-1 below.

Figure 19-1: Copper and Gold Concentrates



Source: Ocean Partners (2024)

In terms of copper and gold grades the proposed Troilus flotation concentrate is comparable to material produced at the Telfer and Boddington mines in Australia. Although these qualities are not regarded as complex, they do have slightly elevated arsenic and fluorine levels compared to the proposed Troilus product.

A significant proportion of the other Cu-Au concentrates in the market are also understood to attract penalties for elevated levels of deleterious elements. These are typically for arsenic (e.g. Chelopech, Mount Carlton and Lepanto) or fluorine (Grasberg, Salobo, OK Tedi). Concentrate from the LaRonde mine in Québec has the highest gold content of the peer group at around 375 g/t but is regarded as complex due to elevated bismuth levels.

Based on the specifications provided, the Troilus flotation concentrate would be regarded as a very clean material with a slightly lower than average copper grade but a significant gold content. It can be assumed with some confidence that a market will readily be found for the proposed 70kdmt/a of production.

19.4 Marketing and Commercial Terms

19.4.1 International Market - Direct Sales to Smelters

A clean gold bearing material such as the Troilus flotation concentrate is likely to be regarded as an attractive feed material by many smelters. As outlined elsewhere in this report, the highest gold payables for such material are likely to be received from smelters in Japan (PPC / Mitsibushi) and Korea (LS Nikko) or from European buyers such as Aurubis or Boliden, with the Chinese market lagging behind mainly due to decreased gold payments.

Direct sales to smelters can be linked to prevailing benchmark TCRCs which for 2024 are $80/t & 8.0 c/lb. Alternatively, a long-term agreement can be made at fixed TCRCs. As discussed elsewhere in this report, it is expected that future long-term average benchmark TCRCs will average around $80/t & 8.0 c/lb in 2023$ terms.

Indicative terms from key potential off-takers are presented in the paragraphs below. These are based on recent discussions and business with the relevant smelters.

It is likely that international smelters will require minimum lot sizes of at least 5kdmt in bulk. Likely receivers including Aurubis, Boliden and Pan Pacific tend to reserve limited container handling capacity for higher grade gold concentrates. As such, it is likely that bulk shipments would be required for the Troilus concentrate. If container shipments were accepted, there are likely to be additional handling charges of $15/wmt for bulk in containers and $30/wmt for bags in containers.

Pan Pacific Copper in Japan has indicated likely terms for purchase of Troilus concentrates as follows in Table 19-6:

Table 19-6: Pan Pacific Copper Indicative Terms

Copper payment	96.5%. Minimum deduction 1.1 units
Silver payment	90% above 30g/t.
Gold payment	90% If over 1g/dmt up to and including 3g/dmt 93% If over 3g/dmt up to and including 3g/dmt 95% If over 5g/dmt up to and including 10 g/dmt 96% If over 10 g/dmt up to and including 15g/dmt 97% If over 15g/dmt up to and including 20g/dmt 97.5% If over 20g/dmt up to and including 20g/dmt 97.75% If over 30g/dmt up to and including 50 g/dmt 98% If over 50 g/dmt up to and including 80 g/dmt 98.15% above 80 g/dmt
Treatment charge	Benchmark. Assume LT Average US$70/t CIFFO Japan
Copper refining charge	US$0.07/lb
Silver refining charge	US$0.4/oz
Gold refining charge	US$4.0/oz

Aurubis has expressed a firm interest in taking 40kdmt/a of production and would be willing to offer an annual benchmark + premium related TCRC or a fixed TCRC for a long-term agreement. The likely terms that could be negotiated with Aurubis for a long-term offtake agreement are summarised in Table 19-7 below.

Table 19-7: Aurubis Indicative Terms

Copper payment	96.6%. Minimum deduction 1.1 unit
Silver payment	97%. Minimum deduction 20g/t
Gold payment	98%. Minimum deduction 2g/t
Treatment charge	Benchmark + 10%. Assume LT average US$77/t CIF Brunsbuttel, Germany
Copper refining charge	US$0.077/lb
Silver refining charge	US$0.5/oz
Gold refining charge	US$5.0/oz

Sales to Boliden smelters are likely to be at similar terms to those offered by Aurubis. However, shipping costs to discharge ports in Sweden or Finland are likely to be around $15/t higher than for delivery into Germany.

19.4.2 International Market – Sales Via Traders

Sales to international smelters via traders are likely to realise lower TCRCs compared to direct sales to smelters recognizing the typical disconnect between spot terms and long-term contracts.

The low levels of deleterious elements in the Troilus flotation concentrate could also make the material attractive to traders for blending with more complex materials. Blending is typically required to meet the overall feed requirements of a particular smelter or to comply with the Chinese import limits.

Traders are also likely to offer significantly earlier payment terms than would be achieved in direct sales to smelters. This could represent an important saving in working capital given the potential need to consolidate cargos at the ports of Montreal or Québec City.

A trader would also be able to realise any potential synergies from consolidating international shipments with output from other local mines such as the proposed Dore Copper Mining Corp project at Chibougamau or LaRonde. Such savings would likely be reflected in the terms offered for purchases from traders.

As a trading company that is active in this market Ocean Partners can offer terms as follows in Table 19-8:

Table 19-8: Ocean Partners Indicative Terms

Copper payment	96.6%. Minimum deduction 1.1 units
Silver payment	90%. Minimum deduction 20g/t
Gold payment	97.5%. Minimum deduction 2g/t
Treatment charge	Benchmark -10% Assume LT Average US$63/t. Basis DAP Montreal.
Copper refining charge	US$0.063/lb
Silver refining charge	US$0.5/oz
Gold refining charge	US$5.0/oz

19.4.3 Local Market

Sales of Troilus flotation concentrate to Glencore's Horne smelter at Rouyn-Noranda, Québec are a viable option. It is understood that Troilus Gold is in direct contact with Glencore regarding a potential offtake agreement.

Based on our understanding of the terms received by other mines for sales to the Horne the TCRCs would be at or around benchmark levels and the gold payment offered would be 1.0 to 2.0% lower depending on grade than what would be attainable in the international market. Typically, the Horne, is only willing to contract on a multi-year basis of 3-5 years for the entire mine's production.

Headline TCRCs and gold payables achieved in sales to the Horne are likely to be less competitive than those available on the international market. However, delivery would be on the basis of DAP Horne and a freight differential would be negotiated, whereby the buyer and seller share the saving in freight costs compared to delivery CIFFO Main European Port. The Horne tends to have long dated payment terms (typically three or four months after the month of delivery) especially for precious metal bearing copper concentrates. This factor will need to be carefully analysed as part of any decision-making process.

Delivery would be on the basis of DAP Horne and a freight differential would be negotiated, whereby the buyer and seller share the saving in freight costs compared to delivery CIFFO Main European Port. Whether or not the freight saving would be adequate to compensate for the higher TCRCs and lower gold payments compared to international sales via traders would depend on the actual terms negotiated. Through our understanding of other direct shipments to the Horne, the freight saving would be shared anywhere from 25:75 to 50:50 basis (miner: smelter) and usually fixed upfront.

19.4.4 Marketing Strategy

Given its geographical location, the Horne smelter would seem to be the 'natural' home for a significant proportion of the Troilus flotation concentrate. Whether or not the purchase terms and more importantly the payment terms offered by Glencore are competitive will depend on how well the specification offered fits into the Horne's overall smelter feed requirements at the time concentrate production starts at Troilus. It is known that similar local concentrates, such as that produced at Agnico Eagle's LaRonde complex are treated successfully at the Horne and it is likely that the same will be true for the Troilus material. It is understood that the Horne's typical payment terms are 3 to 4 months after the month of delivery to the Horne so selling 100% of material there would likely require significant working capital.

It would be possible for Troilus to sell material directly to Glencore for processing at the Horne smelter. Such sales are likely to be through a long-term 'frame' contract under which matters such as metal payments and delivery schedules would be fixed, but commercial terms including TCRCs, and any freight differential would be subject to periodic re-negotiation. In these circumstances, it would be necessary to ensure that the purchase terms offered by Glencore remain market competitive. Annual renegotiation of the contract terms, tonnage ranges at the Seller's option and engaging a consultant or agent to manage the sales may present ways of mitigating this risk. It is highly suggested that if Troilus wants to commence negotiations with Glencore that it engage an agent or market advisor that is active in international markets for Au/Cu concentrates (rather than a consultant who does not actively trade) as the Au/Cu markets are small and dynamics change very quickly.

Troilus could also consider reserving a portion or range of the planned annual production for annual or spot sales to traders or other smelters. This strategy would allow for some flexibility around production schedules. Spot sales to traders or other smelters would provide a reliable comparison between the terms received under any long-term contract with Glencore and prevailing rates on the international market. In addition. when negotiating with Glencore for the commercial terms associated with sales to the Horne smelter it would be advantageous to be able to demonstrate that there are other realistic off-takers for the Troilus concentrate.

In addition to likely attracting lower TCRCs and importantly prompt payment terms to reduce working capital, conducting any international business through traders rather than via direct sales to other smelters would likely reduce the additional administration and logistics that Troilus Gold Corp would have to handle. A trader would also be able to realise any potential synergies from consolidating international shipments with output from other local mines such as the proposed Dore Copper Mining Corp project at Chibougamau or LaRonde as examples. Such savings would likely be reflected in the terms offered for purchases from traders.

19.4.5 Finance Options

Typically, traders offer earlier payment terms than would be offered in direct sales to smelters. Troilus will therefore be presented with opportunities to conserve working capital.

It is understood that payment for deliveries to the Horne will typically be 3MAMD or 4MAMD (three or four months after the month of deliveries). Although truck shipments to the Horne will eventually generate a regular monthly cashflow for Troilus, there will be a significant working capital portion that would need to be financed upfront and essentially for the life of mine. It is also worth highlighting that if commodity prices rise, then more working capital financing is required and conversely should prices go lower.

Early payment from a trader would be a significant advantage for any international sales given the longer shipping time involved and the potential need to consolidate cargos at the ports of Montreal or Québec City.

Other international smelters tend to have payment terms based on arrival (20 – 45 days after arrival) so if selling direct to other international smelters would need to factor in the additional ocean voyage and payment terms when comparing the Horne or a trader's payment terms.

Another option that could be explored is the possibility of securing a portion of project finance against a long-term offtake agreement with a major smelter. For example, Aurubis is known to work with the KfW IPEX-Bank to secure feed material (recent examples include Marcobre and Fruta del Norte) which it regards as being favourable for its operations. Korean, Japanese and other European import/export banks also support offtake linked financings although not as prevalently as Aurubis. Glencore in its regular trading business does provide project financings to mining projects, however this is typically something that is not provided to specifically secure feeds for the Horne which tends to rely on its geographic advantages.

19.4.6 Freight Costs and Logistics

At the time of writing, prevailing rates in the dry bulk freight and container markets have retreated from the multi-year highs reached during 2021 and are now below long-term averages.

We understand the normal trucking rates from the mine site to the ports of Montreal or Québec City are between $60 to $90 per wmt. Over the long-term rates should trend towards this long- term average level. Port costs are typically between $15 and $20 per wmt.

It is expected that sea freight rates will remain slightly higher than historic long-term averages reflecting the higher price of fuel and the cost implications of new environmental regulations relating to the shipping industry. In normalized markets bulk sea freight costs to Europe may revert to $45 to $65 per wmt and $65 to $90 per wmt to Asia.

Typically, container markets in normalized markets trade at slightly cheaper rates than bulk however most international Asian and European smelters have very strict tonnage limits on deliveries they will take in containers and bags, and we do not expect that any single smelter would be willing to receive 70,000 tonnes per annum in bags or bulk in containers. Shipping in containers adds significant amounts of contract administration and logistics work (rather than 8 bulk shipments a year likely 52 weekly shipments per annum).

Outside of the tonnage limits, given the additional handling requirements for containers international smelters tend to apply a $15 per wmt per bulk in container surcharge and a $30 per wmt per bag in container surcharge.

Given the recent volatility in rates, freight must be watched very carefully when making any long-term sales decisions.

19.4.7 Commercial Terms

Commercial terms for the options described are summarised in Table 19-9 below.

Table 19-9: Commercial Terms (LOM Average)

	Direct sales to Aurubis	Direct sales to Boliden	Direct sales to PPC (Japan)	Sales to Ocean Partners
Delivery	CIF FO Brunsbuttel, Germany	CIF FO Skelleftea, Sweden	CIF FO Saganoseki, Japan	DAP Montreal warehouse.
Inland truck freight	$72.5/wmt	$72.5/wmt	$72.5/wmt	$72.5/wmt
Port handling costs	$20/wmt	$20/wmt	$20/wmt	Quarterly freight credit to be agreed equivalent to CIF FO Taiwan basis 5,000 wmt shipments for Seller's account. ($110/wmt based on assumed freight rates to Asia).
Ocean Freight	$65/wmt	$80/wmt	$90/wmt	
Freight adjustment				
Treatment charge	Benchmark + 10%. Assume $77/dmt	$85/dmt	Benchmark. Assume LT Average $70/dmt	Benchmark -10% Assume LT Average $63/dmt
Copper payment	96.6% MD 1.1 units	96.6% MD 1.1 units	96.5% MD 1.1 units	96.6% MD 1.1 units
Copper refining charge	$0.077/lb	$0.85/lb	$0.07.lb	$0.063/lb.
Gold payment	98% MD 2g/t	97% MD 2g/t	98.15%	97.5% MD 2g/t.
Gold refining charge	$5.0/oz	$5.0/oz	$4.0/oz	$5.0/oz
Silver payment	97% MD 20g/t	97% MD 20g/t	90%	90%
Silver Refining charge	$0.50/oz	$0.50/oz	$0.40/oz	$0.50/oz
Penalties	Assumed to be a clean concentrate attracting no penalties.			
Estimated Ocean Transit Time	30	45	75	75
QP	2 MAMA	2 MAMA	2 MAMA	M+1 or 2 MAMA with option to price at any point before onset of QP.
Payment	30 days after arrival	10 days after arrival	30 days after arrival	End of Month after month of delivery to warehouse. Earlier financing available at 3M LIBOR + 5.5%.

The treatment and copper refining charges and freight costs presented above are long-term averages intended for financial modelling purposes. As described in the discussion of the global copper concentrate market included in this report, TCRCs are likely to be declining to levels below long-term averages in the years immediately following the proposed project start date in 2024. Likewise, freight costs may not have fully corrected from current elevated levels by that time.

19.5 Global Copper Concentrate Market

After a two-year period during which TCRCs had been relatively high, reflecting the ready availability of material, the dynamics of the global copper concentrate market changed during the second half of 2023. Demand for copper concentrate from new smelting projects started to outpace global mine production resulting in a decline in spot terms and the annual benchmark for 2024 being established at $80/t & 8.0 c/lb, compared to $88/t & 8.8 c/lb in 2023.

Downward momentum in spot terms accelerated during the early months of 2024 following the unexpected closure of the Cobre Panama mine. Spot TCRCs fell to multi-year lows, close to or below $0/t & 0.0 c/lb. This dramatic fall reflected a growing realisation that the global copper concentrate market had entered an extended period of tightness combined with a significant contango in the refined copper price that enabled traders to bid sharp terms for material. Spot terms this low clearly put severe pressure on smelter economics and are lower than would appear to be justified by the relatively modest deficit that is forecast for H1 2024. Some correction is likely as smelters scale back utilisation rates, but TCRCs are expected to remain lower than long-term averages for an extended period during the second half of this decade.

The catalyst for this change in market dynamics has been the construction of several new smelters outside China. In Indonesia, the 1.7Mdmt/a Manyar smelter and 900Mdmt/a PTMANT project are both expected to be commissioned in 2024 and ramp up towards full capacity in 2025. The Kamoa-Kakula smelter (500kt/a of anode) in DRC is reportedly on target for completion by late 2024 and reports in India now suggest that Phase 1 (500kt/a) of a new smelter being constructed by Adani may also fire up towards the end of 2024.

Increases in Chinese demand which have been the dominant force in the global copper concentrate market for the last 20-years are also likely to continue. Five new Chinese smelter projects are currently scheduled to come online during 2024. These are South-West Copper's 350kt/a side-blown furnace at Yunnan, Fubang's new 50kt/a side-blown furnace in Inner Mongolia, an 85kt/a side-blown furnace being constructed by Weihai Humon Chemical in Shandong province and two 350kt/a projects being built by Jinchuan Group. A shortage of feed materials could potentially lead to one or more of these projects being delayed. However, it should be noted that despite the rapid growth in Chinese copper smelting capacity over the past decade, the country continues to import over 3Mt/a of refined copper. Under these circumstances the incentive for further growth in the countries smelting capacity remains, as regional and national governments seek to maximize local 'value-added.'

The commissioning during 2023 of Teck Resources Quebrada Blanca project in Chile marks the end of the recent wave of large greenfield copper mine construction projects. The gradual ramp up of projects such as Oyu Tolgoi underground and Chuquicamata underground and brownfield expansions such as those at Mantos Copper will add some incremental capacity in the coming years. However, output of copper in concentrate from existing copper mines is expected to peak by around the middle of this decade due to declining grades and depletion of ore reserves.

Mining companies have been generally reluctant to commit to new copper mine construction projects over recent years. Even in the current market with the copper price around $10,000/t, the major miners seem more focused on merger and acquisition activity than organic growth.

New copper mining projects tend to require a large throughput to realise the necessary economies of scale required to profitably process lower grade ore bodies. Additionally, projects frequently require significant investment in infrastructure such as port facilities or seawater desalination plants. As such, even if new projects are approved promptly, the extended construction period required to bring them online probably mean that they will have little impact on the expected tightness on the global copper concentrate market during the second half of this decade.

Miners need smelters to remain profitable so that they have a market for their production. A baseload of smelter feed secured at reasonable terms under long-term contracts is therefore necessary for the health of the industry. After allowing for inflation, benchmark TCRCs have averaged around $97/t and 9.7 c/lb in 2024$ over the past decade.

Given the expected structural tightness of the copper concentrate market outlined in the preceding paragraphs and the fact that benchmark terms peaked below $90/t & 9.0 c/lb in the most recent cycle, the long-term average benchmark TCRC is likely to be lower than the inflation adjusted figure quoted above. On this basis, a long-term benchmark TCRC of $70/t & 7.0 c/lb is thought to be a reasonable assumption for financial modelling purposes. This can be regarded as a conservative assumption reflecting the desire of financing banks to see calculations based on direct sales to smelters under long-term contracts. It is probable that at least a portion of the Project's output will be sold to traders under either long-term or spot contracts with realized TCRCs at least 10% lower than the current benchmark.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Background

The Site was previously operated from 1998 to 2011 and was partially rehabilitated from 2011 in accordance with the closure plan approved by MERN for the closure on the Mine. The operating history provides an advantage of having real data from which to assess the impacts and effects of future exploitation with precision.

The Site has currently two environmental statuses: exploration and closed (reclaimed) sites. Site reclamation commenced at the end of the previous operation. Buildings and infrastructures have been dismantled. Soils have been characterized and reclaimed. The waste rock piles, and the tailings pond have been revegetated. The remaining work for closure consists of removing the pumps from the tailings pond and having the water flow naturally via a channel to the receiving environment. The exploration status relates to the drilling and finding new resources for an eventual restart of the operation.

In November 2019, the Company submitted an environmental impact assessment to MELCC (Ministère de l'Environnement et de la Lutte contre les Changements Climatiques du Québec) for the dewatering of the J4 and 87 pits at the Property. The Company engaged in community consultations with impacted families on the Property and the local communities of Mistissini and Chibougamau to keep them informed of the dewatering proposal and integrate the feedback of stakeholders. In August 2020, the Company received a Certificate of Authorization from MELCC to proceed with dewatering.

Dewatering the pits is expected to take 1 to 2 years and will allow the Company to access drilling targets that are currently underwater to continue exploration of the property. Infrastructure to support the dewatering, such as a water treatment and pumping facility, have been installed at site in 2020. The pumping will start at the end of summer 2024.

20.2 Regulatory Context and Permitting

20.2.1 Federal

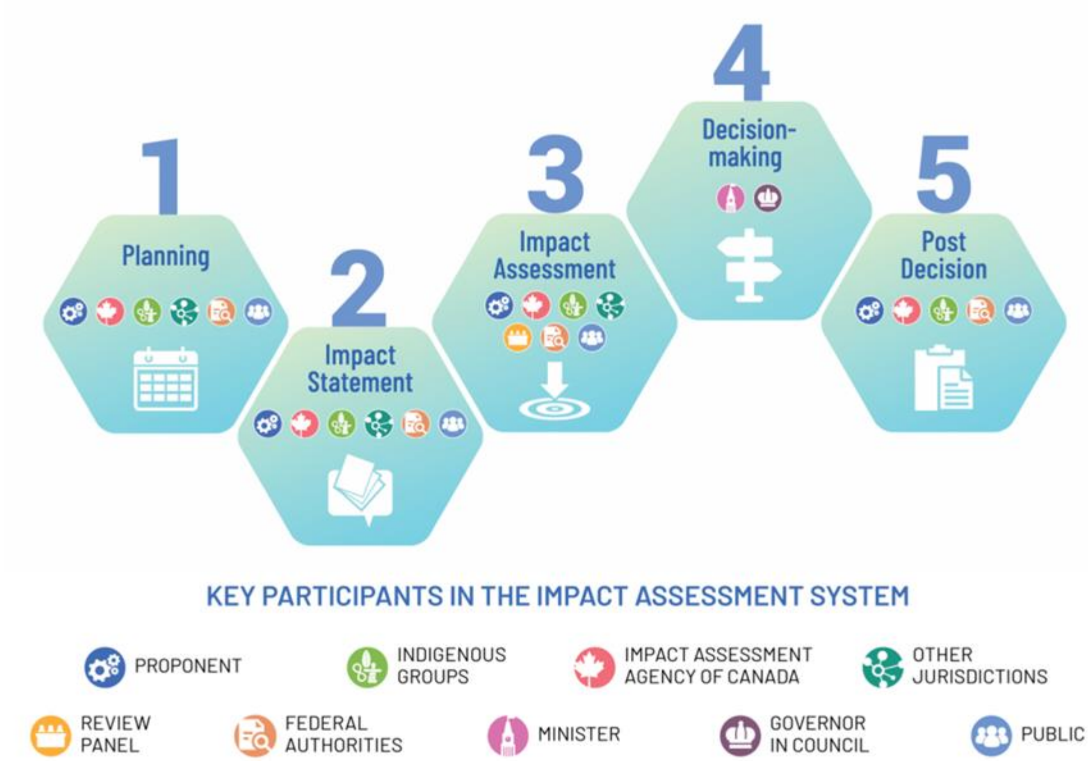
Under the Impact Assessment Act (IAA 2019), only projects designated by the Regulations Designating Physical Activities (DORS/2019-285) are subjected to the environmental assessment procedure. Thus, an environmental assessment under the IAA 2019 is required for a project that involves the construction, operation, decommissioning and abandonment of a new copper/gold mine, other than a placer mine, with an ore production capacity of 5,000 t/day or more.

Troilus is planning to open a mine with an ore production capacity more than 5,000 t/day and is thus subjected to the IAA. The process has started with the filing of an Initial Project Description in May 2022 and is in the Planning Phase (reference). The entire process is a 5-steps process as shown in Figure 20-1. It should not take more than 3 to 4 years to go through the Phase 1 to the Phase 5.

The link to the Troilus Project of IAAC is:

- https://aeic-iaac.gc.ca/050/evaluations/proj/83658?culture=en-CA

Figure 20-1: Federal Impact Assessment Process Overview



https://www.canada.ca/en/impact-assessment-agency/services/policy-guidance/impact-assessment-process-overview.html
Source: Troilus Gold

On October 13, 2023, the Supreme Court of Canada ruled that the Impact Assessment Act was largely unconstitutional. The Parliament of Canada must adjust its legislation to bring it into line with this judgment and continue to assess and regulate the federal aspects of projects. According to the Supreme Court, the federal law is too broad. Based on the recommendations of the Impact Assessment Agency of Canada, Troilus Gold is continuing the process in place to ensure that there is no delay in the completion schedule. Subsequent amendments to the Act would not delay the project or increase the already stringent requirements of the federal jurisdiction aspects.

20.2.2 Provincial

The opening and operation of a mine triggers the environmental impact assessment and review procedure under chapter II of the Environment Quality Act (EQA). This chapter covers the particular regime defined by the James Bay and Northern Québec Agreement (JBNQA). The process includes a participation by the First Nations groups so that they can protect the rights and guarantees granted to them under the Agreement.

Under the JBNQA, the James Bay Advisory Committee on the Environment (JBACE) was created for projects south of 55th parallel. To evaluate and review development projects within the jurisdiction of Québec, two other committees were created:

- The Evaluating Committee (COMEV), a Québec / Cree / Canada agency responsible for assessing and drawing up guidelines for the impact study of projects located south of the 55th parallel.
- The Review Committee (COMEX): a Québec / Cree agency responsible for reviewing projects south of 55th parallel.

The opening and operation of a copper/gold mine triggers the environmental impact assessment and review procedure under chapter II of the EQA. The process will include five principal steps:

1) preparation and submission of a project notice to the provincial administrator
2) reception of the guidelines
3) preparation and submission of the ESIA
4) review and recommendation by the Administrator
5) delivery of the authorization

The provincial review process will be made jointly by the Cree Authority and to the Government of Québec for the main Certificate of Authorization, according to paragraph 164 of the LQE.

Following that, an operation certificate of Authorization must be obtained from the regional office of the MELCC.

20.2.3 Other Permitting Requirements

The provincial Mining Act provides the framework for the mining lease, the closure plan, and the financial guarantee associated with the closure plan. The mining lease is required to extract ore. To obtain the mining lease, a closure plan must have been submitted to the Ministry of Energy and Natural Resources and approved. In the case of Troilus Gold, a mining lease is still valid for the operation of J4 and 87 pits. An updated mining lease will have to be filed to cover the new resources to be exploited. A new closure plan will have to be submitted for the Project, and once accepted, the current one will become obsolete.

20.3 Environmental Studies

Troilus Gold initiated in 2019 a series of environmental studies to understand the environmental situation and constraints that are required to understand both the physical components (surface water, groundwater, air, noise, soils) and biological environments (fauna and flora) and include an evaluation of impacts with proposed mitigation measures.

The environmental studies focused on a description of existing conditions considering the past activities of the mine. The impact evaluation will consider the presence of existing conditions, as the site has been mostly reclaimed, and new habitats have been created since the cessation of activities.

20.3.1 Geomorphology and Topography

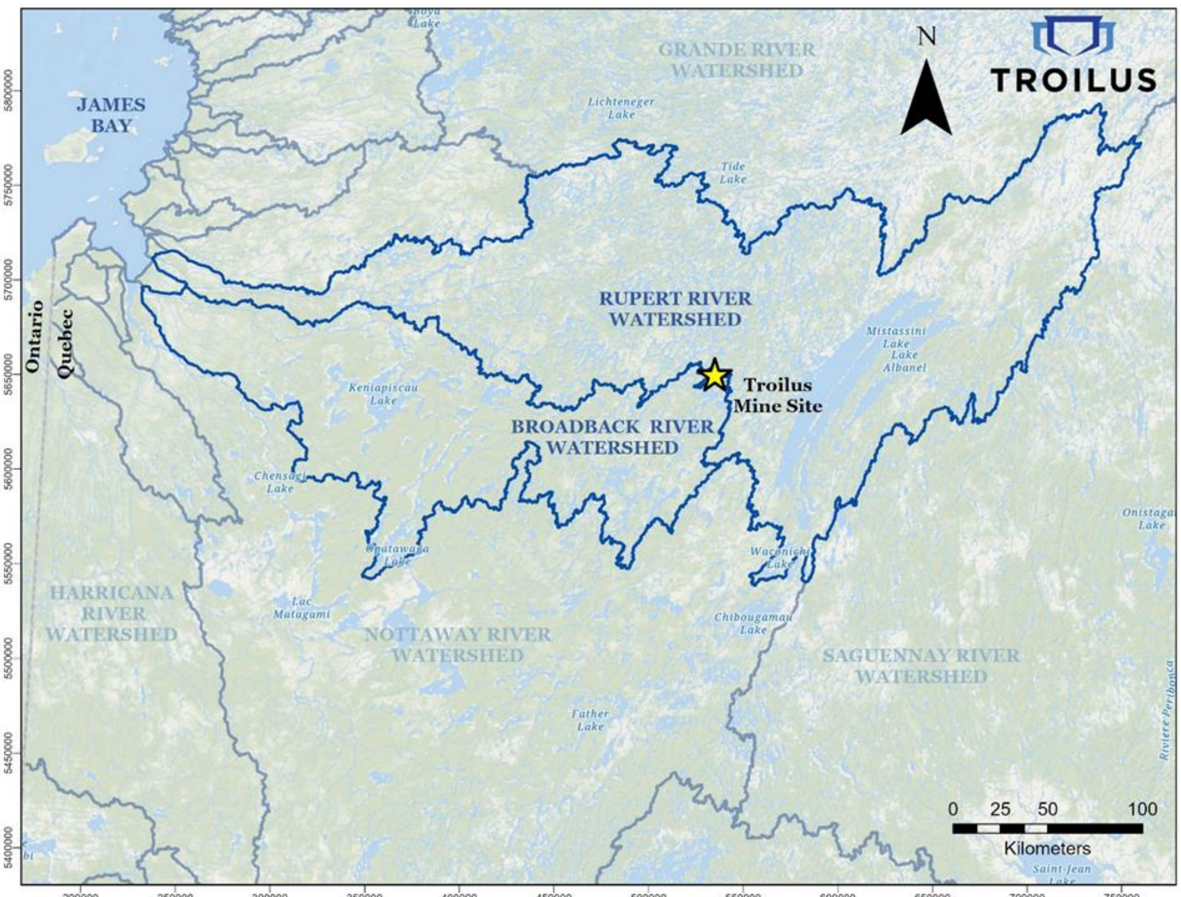
The Troilus mining project site is part of the Eastmain Lowlands, Division of the James Region Physiographic Unit. The terrain is rugged and to the south of the project site, there are rocky hills aligned in a northeast / southwest axis with a maximum altitude of 520 m, while to the north, there is a rocky ridge-oriented northeast / southwest with a maximum altitude of 430 m. Between these two reliefs, the sector forms a valley whose altitude varies between 365 and 400 m with an undulating and irregular topography given the presence of rocky outcrops that intersect the unconsolidated deposits.

20.3.2 Environment Baseline Studies

Hydrology

The Project is located in the Rupert River watershed which drains an area of 43 253 km^2 and reports to the Baie de Rupert and Baie de James (see Figure 20-2). The site of the project is very close to the limit of the Broadback watershed which also reports to Baie de Rupert.

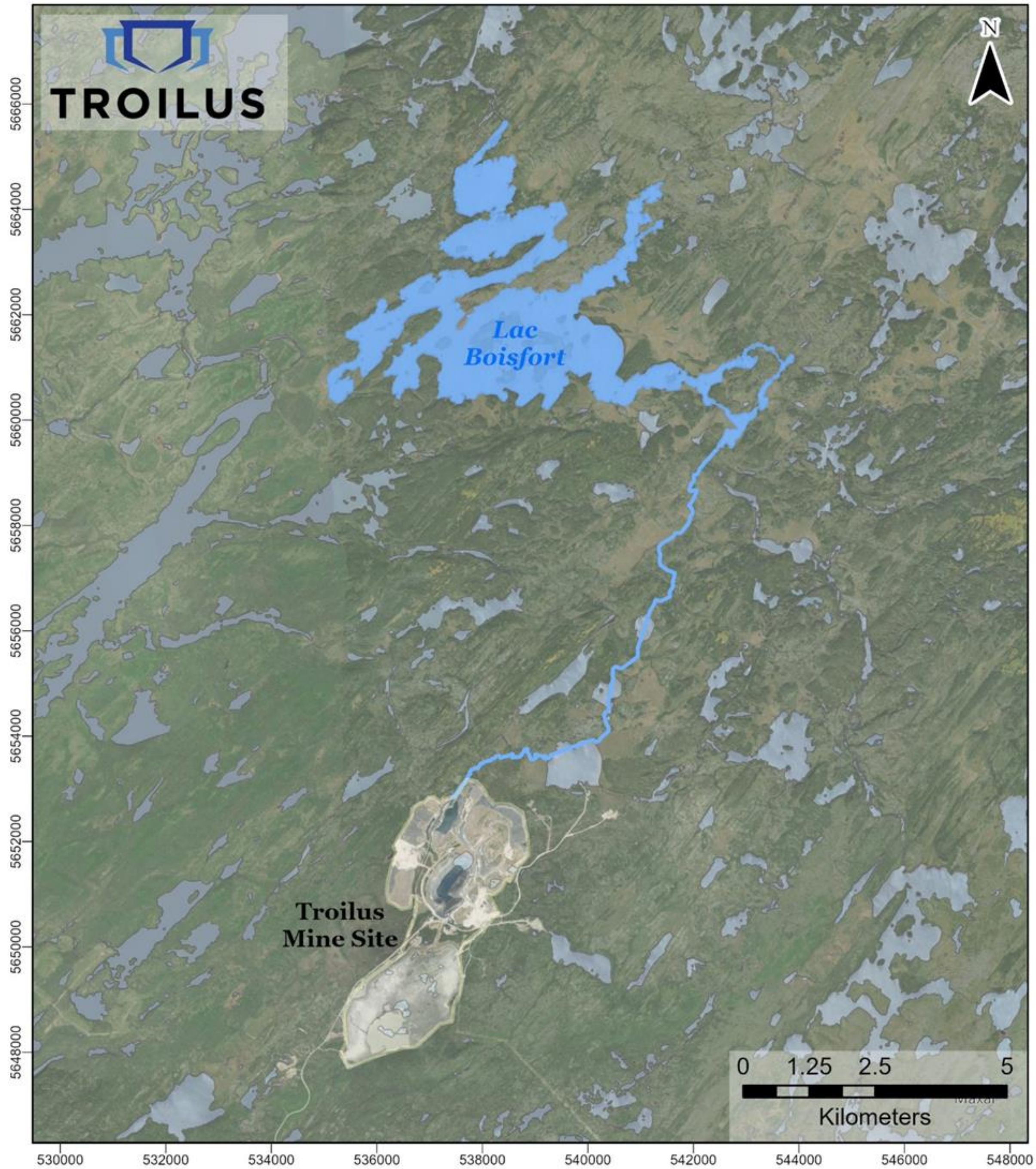
Figure 20-2: Main Watersheds



Source: Troilus Gold

More specifically, the site is located in the Lac Boisfort sub-basin (see Figure 20-3). The site water reports (flow into) the Bibou creek which go through a series of small lakes before reporting, 12 km downstream, to an affluent of Lac Boisfort.

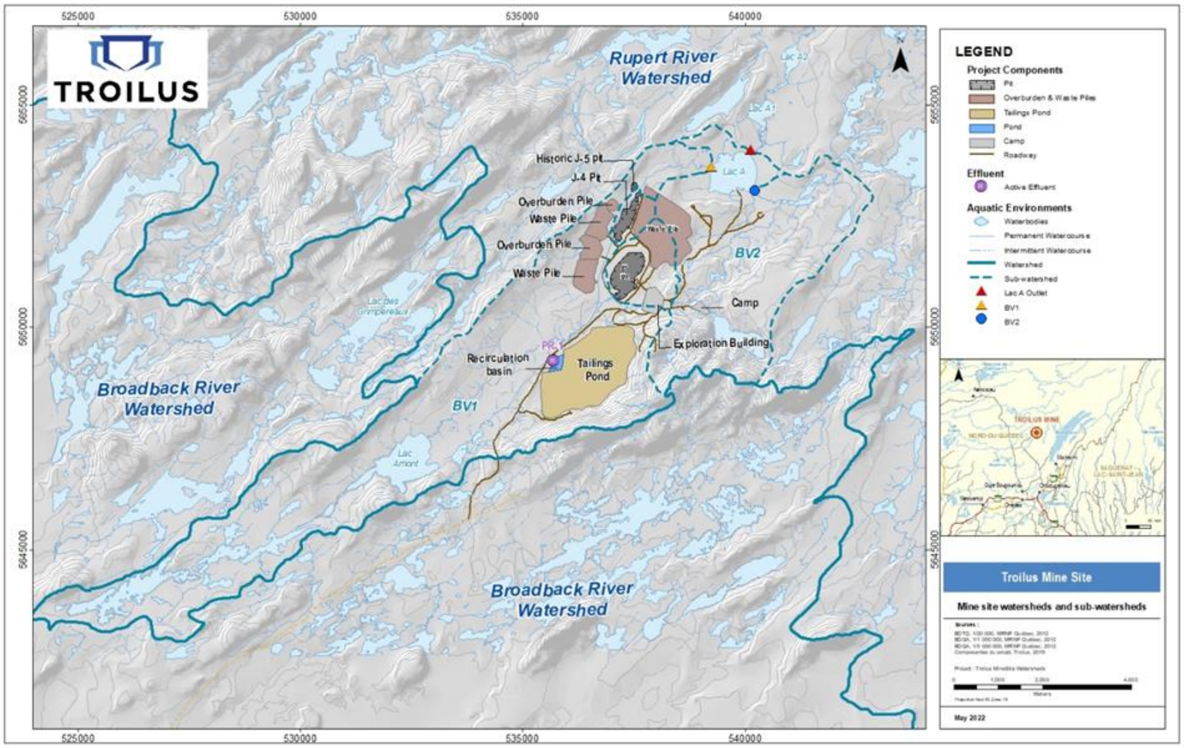
Figure 20-3: Hydrography Downstream the Mine Site



Source: Troilus Gold

At a more detailed level, the mine site is surrounded by two exclusive tributary sub-watersheds of Lake A (BV1 and BV2) (Hydro-Resources Inc., 2019). BV1 is the main watershed and covers an area of 40 km^2. BV2 covers an area of 15 km^2 south of Lake A (Figure 20-4).

Figure 20-4: Site Watersheds



Source: Troilus Gold

The watershed at the outlet of Lake A, which receives water from the Site, covers an area of approximately 58 km^2. The outlet of Lake A is the outlet of the hydrological network of the valley where the Troilus mining project is located.

Spring floods occur during the months of May and June. This flood period can represent 33% of the annual flow. The low water level occurs in winter, between January and April. During this period, the flow can represent less than 8% of the annual flow (Troilus Gold Corp., 2019).

Surface water quality monitoring was carried out following the end of mining operations at certain locations on the site. It is interesting to note that naturally, the hardness of the water is very low (Troilus Gold Corp., 2019). It seems that the presence of the mining site brings elements that increase the hardness of the water, in particular a concentration of calcium and magnesium minerals. The water quality at the mining site is currently affected by the presence of waste piles and certain criteria are beyond the surface water quality criteria for the protection of aquatic life (chronic effect). This is not a concern as it is normal for a brownfield site and will be addressed in the coming impact study.

Hydrogeology

Two distinct geological units corresponding to two hydrogeological units are observed at the site, namely till and rock (Troilus Gold Corp., 2019). The till unit can be subdivided into two sub-units, namely the surface sand, the average grain size of which corresponds to fine to coarse sand with a little gravel, and the till itself, the average grain size of which corresponds to fine sand to medium silty with some gravel.

The hydraulic conductivities of the main subunits are as follows:

- sand subunit, average thickness of 4.0 m and median hydraulic conductivity of 1.7×10^{-5} m/s
- till subunit, average thickness of 6.0 m and median hydraulic conductivity of 6.8×10^{-6} m/s
- rock unit within a distance of 1,000 m from the walls of pit 87, median hydraulic conductivity of 6.0×10^{-7} m/s
- rock unit more than 1,000 m from the pits, median hydraulic conductivity of 3.7×10^{-7} m/s

Before starting the previous mine, piezometry showed an overall northerly flow direction following the surface flow network. In the area of the actual drained pits, the flow was oriented towards the northeast with a gradient of 0.5 to 1%. The dewatering of pit 87 seems to have gradually influenced the piezometry of the surface aquifer; the elevation of the water in the rock aquifer was lowered by 35 to 100 m depending on the location during the operation.

Several observation wells have been monitored since the closure of the Troilus mine site. They are located upstream and downstream of the tailings pond; upstream and downstream of the mine site; in the former industrial sector; as well as downstream of the trench sanitary landfill site. The well used by the Awashish family on the east side of Lake A is also monitored to ensure the quality of drinking water.

The semi-annual monitoring (May and September) of groundwater at the Troilus mining project site reveals the following main facts:

- groundwater upstream of the tailings facility is naturally acidic (pH less than 6)
- conductivity of the water upstream of the site is less than 100 µmhos/cm while that downstream of the tailings pond is between 100 and 400 µmhos/cm and that in the former industrial sector is between 200 and 450 µmhos /cm
- groundwater upstream of the site is of good quality and does not present any contaminants compared with the provincial criteria for groundwater
- dissolved copper concentration is close to the allowable limit for groundwater resurgence criteria (7.3 ppb) in the former industrial sector. In the recent Baseline, concentrations of 10 ppb are present at this location; this is expected from a brownfield site especially because underground water is in contact with mineralized rock enriched with copper and will be closely monitored in the future operation. In other areas, underground water quality is meeting all criteria from MELCCFP
- all the observation wells show values below the detection limit for polycyclic aromatic PAH hydrocarbons during the last sampling campaigns

Vegetation and Wetlands

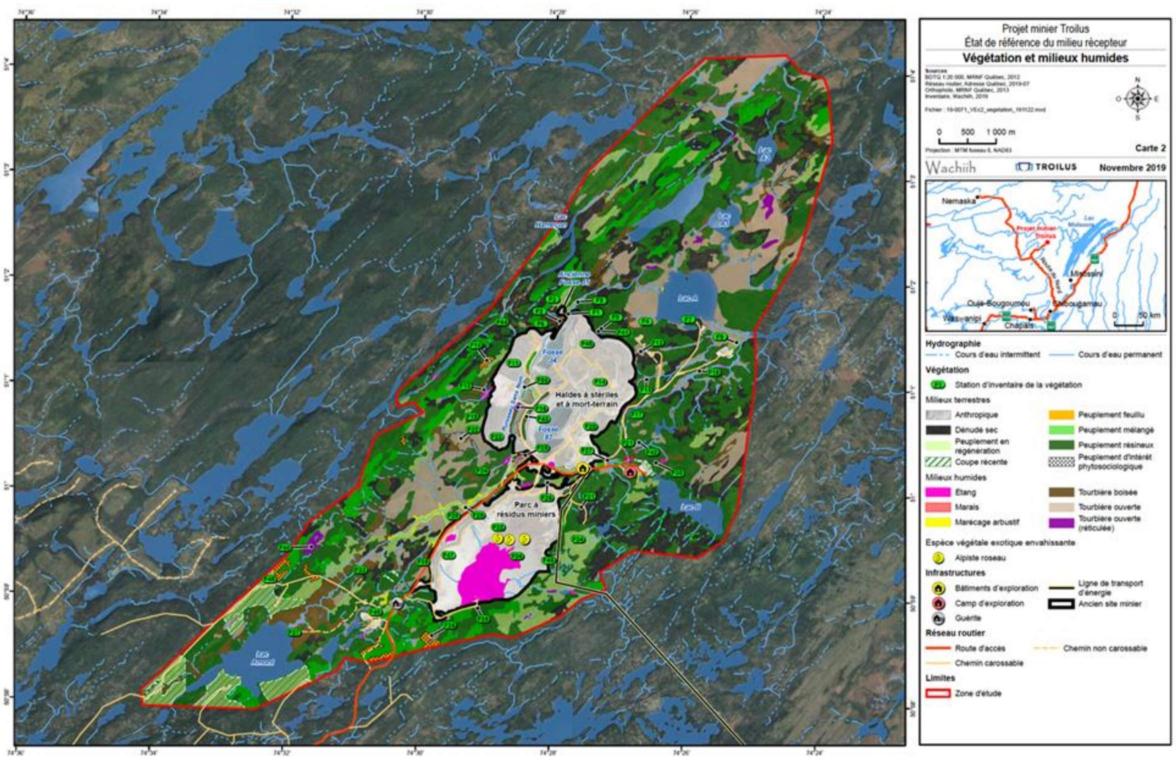
The Troilus mining project is located in the boreal vegetation zone, and more specifically in the continuous boreal forest subzone. The project site is also located in the spruce-moss bioclimatic domain, West sub-domain (Wachiih, 2019a).

The main tree species present in the project area are jack pine (Pinus banksiana) and black spruce (Picea mariana). Other species are also present, but with lower densities, namely paper birch (Betula papyrifera var. papyrifera), trembling aspen (Populus tremuloides) and tamarack (Larix laricina). It should be noted that logging has been carried out in recent years south of Lake Amont (Wachiih, 2019a).

During an inventory carried out in 2019 on the mine site, only one invasive alien plant species (IAS), namely the reed canarygrass (Phalaris arundinacea), was observed in the area of the tailings pound (Wachiih, 2019a). The species was scarce, and no other IAS had been detected during the inventory.

Wetlands are present in the project area. These are mostly open bogs, wooded bogs, ponds, marshes, and shrub swamps (Wachiih, 2019a). During the 2019 inventory, two large peat bog complexes were identified, one between Lake Amont and the sector of the waste rock piles and the other in the sector of lakes A, A1 and A2 (Wachiih, 2019a). Figure 20-5 illustrates the wetlands near the Project.

Figure 20-5: Vegetation and Wetlands in Lake A , A1 and A2 Watershed



Source: Wachiih

Following the consultation of the Data Center on Natural Heritage of Québec (CDPNQ) in 2019, no occurrence of special-status plant species was reported on the site or nearby (Troilus Gold, 2019). Coming field work will make sure to include looking for those species and report any observation.

It should be noted that certain sectors and infrastructures used during previous mining operations (waste rock pile, tailings site and industrial sector) are currently being restored and have been mostly seeded with the following species: grasses, clovers and bird's-foot trefoil. There are also a few shrubs, hardwood, and pines less than one meter tall in the mine restored areas.

Fish and Fish Habitat

Field inventories were carried out in 2018 and 2019 to characterize certain waterbodies located on the project site, i.e. unnamed creeks, lakes A, A1, A2, and B as well as the watercourses connecting these lakes.

Field inventories are planned in the coming months to complete the characterization of all the waterbodies that could be impacted by the Project.

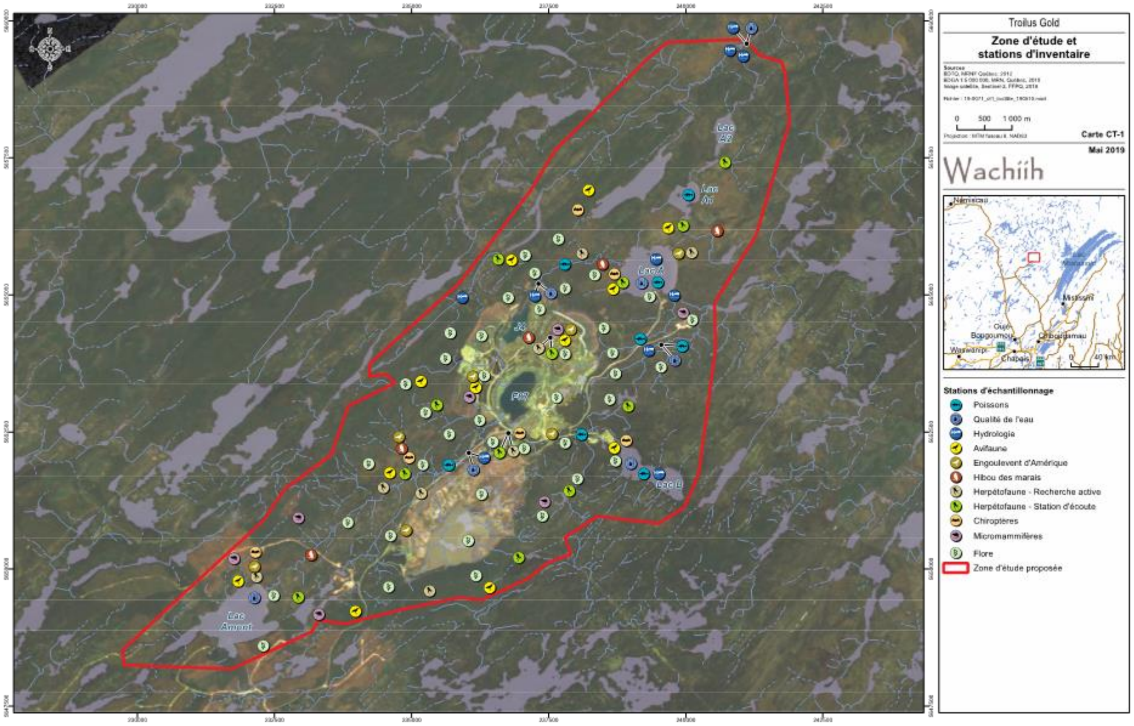
Fish were captured during these inventories in the water bodies, representing the following eight species: lake cisco (Coregonus artedi), walleye (Sander vitreus), northern pike (Esox lucius), lake whitefish (Coregonus clupeaformis), white sucker (Catostomus commersonii), longnose dace (Rhinichthys cataractae), brook trout (Salvelinus fontinalis) and yellow perch (Perca flavescens) (Wachiih, 2019b). Lake A is the body of water with the greatest diversity of species caught. The dominant species in Lake A is walleye, followed by lake cisco. In Lake A1, lake whitefish largely dominate the catches, followed by walleye. In Lake B, only northern pike and white sucker were caught. Lake cisco and yellow perch were only captured in lakes A and A1, respectively. The northern pike is the only species common to the three bodies of water.

A total of 26 fish were caught in the creeks and river during the 2018 and 2019 inventories. The only two species caught were longnose dace and brook trout.

Finally, a total of 19 spawning grounds were identified during these inventories (Figure 20-6). Of this number, three of them would be used by walleye or by suckers for spawning (Wachiih, 2019b).

Bibou Creek is planned to be relocated north-west of the infrastructures.

Figure 20-6: Birds and Mammals – Terrestrial Fauna



Source: Wachiih

During the avian inventory carried out in 2019, 62 different avian species were observed in the area of the Troilus mining project (Wachiih and FaunENord, 2019a). This inventory confirmed the presence of four species at risk in the sector, namely the common nighthawk (Chordeiles minor), the short-eared owl (Asio flammeus), the olive-sided flycatcher (Contopus cooperi) and bank swallow (Riparia riparia).

Acoustic inventories of bats were carried out in 2019 in the area of the Troilus mining project. A total of 102 sonograms were collected and of these, only one contained chiropteran echolocation calls (Wachiih and FaunENord, 2019c). The only species identified is the big brown bat (Eptesicus fuscus). No concentration of individuals (e.g. maternity) was detected following these sound analyses.

Inventories of herpetofauna were also carried out in 2019 in the area of the Troilus mining project. These inventories included listening to the songs of anurans as well as the active search for amphibians and reptiles. In total, seven different species of herpetofauna have been identified. More specifically, four species of anurans, spring peeper (Pseudacris crucifer), American toad (Anaxyrus americanus), wood frog (Lithobates sylvaticus) and northern frog (Lithobates septentrionalis), two species of urodeles, namely the two-lined salamander (Eurycea bislineata) and the blue-spotted salamander (Ambystoma laterale) as well as a species of reptile, namely the garter snake (Thamnophis sirtalis) have been listed. The spring peeper is the species most often observed in the territory during surveys (Wachiih and FaunENord, 2019b).

An inventory of micromammals was carried out in 2019 on the Troilus mining project site. A total of 120 specimens of nine different species of micromammals were captured as part of this sampling campaign (Wachiih and FaunENord, 2019c). The red-backed vole (Myodes gapperi) and the common shrew (Sorex cinereus) are the two most abundant species on the territory, and the Southern bog lemming (Synaptomys cooperi) an endangered specie is present on the site. Mitigation measures have been presented in the dewatering impact study (Troilus Gold Corp. 2019).

In addition, an inventory of large fauna was carried out in 2019 in the area of the Troilus mining project. The large wildlife species targeted were woodland caribou (Rangifer tarandus caribou), moose (Alces alces), black bear (Ursus americanus) and gray wolf (Canis lupus). The inventory made it possible to locate and classify 109 caribou, i.e. 11 males, 15 females, 9 calves and 74 undetermined (WSP, 2019). These results correspond to a density of about 5.97 caribou per 100 km². Range collar occurrence points demonstrate that caribou use the area throughout their annual life cycle. The aerial inventory made it possible to locate 16 track networks corresponding to moose wintering areas in the sector. In the moose survey area, a total of three individuals (a female, a calf and an indeterminate) were observed in two wintering (ravaging) areas, which corresponds to an estimated density of 0.40 moose/10 km² (WSP 2019). Although no specific inventory has been carried out for the black bear, the species is present throughout the territory according to the families consulted. It should be noted that the species frequents the landfill site on the mine site and the tailings site where the herbaceous layer is abundant. During the inventory, two networks of wolf tracks were detected. The presence of the species on the territory was also confirmed by some of the families consulted.

There are no protected areas in the vicinity of the Troilus mining project.

Endangered Wildlife

The fauna and flora species at risk potentially present in the project area are presented in Table 20-1.

Table 20-1: Fauna and Flora Species at Risk Potentially Present in the Project Area

English Name	Latin Name	Québec Status	Canada Status
	Oiseaux		
Golden eagle	*Aquila chrysaetos*	Vulnerable	Not at risk
Yellow rail	*Coturnicops noveboracensis*	Threatened	Special Concern
Short-eared owl	*Asio flammeus*	ESDMV	Threatened
Narrow-billed Phalarope	*Phalaropus lobatus*	-	Special Concern
Wandering grosbeak	*Coccothraustes vespertinus*	-	Special Concern
Hirondelle rustique	*Hirundo rustica*	-	Special Concern
Harlequin Duck, Eastern population	*Histrionicus histrionicus*	Vulnerable	Special Concern
Bald Eagle	*Haliaeetus leucocephalus*	Vulnerable	Not at risk
Common Nighthawk	*Chordeiles minor*	ESDMV	Special Concern
Olive-sided Flycatcher	*Contopus cooperi*	ESDMV	Special Concern
Canada Warbler	*Cardellina canadensis*	ESDMV	Special Concern
Rusty Blackbird	*Euphagus carolinus*	ESDMV	Special Concern
Common Nighthawk	*Chordeiles minor*	ESDMV	Special Concern
Bank Swallow	*Riparia riparia*	-	Threatened

English Name	Latin Name	Québec Status	Canada Status
Fish			
Yellow Sturgeon	*Acipenser fulvescens*	ESDMV	Endangered
Mammals			
Woodland caribou, forest ecotype	*Rangifer tarandus caribou*	Vulnerable	Threatened
Pygmy weasel	*Mustela nivalis*	ESDMV	-
Rock vole	*Microtus chrotorrhinus*	ESDMV	-
Cooper's lemming vole	*Synaptomys cooperi*	ESDMV	-
Silver bat	*Lasionycteris noctivagans*	ESDMV	-
Gray bat	*Lasiurus cinereus*	ESDMV	-
Red bat	*Lasiurus borealis*	ESDMV	-
Little brow Myotis	*Myotis lucifugus*	-	Endangered
Northern Myotis	*Myotis septentrionali*	-	Endangered
Eastern wolf	*Canis sp. cf. lycaon*	-	Threatened
Plants			
American Calypso	*Calypso bulbosa*	ESDMV	-
Shrub Willow	*Salix arbusculoides*	ESDMV	-
McCalla willow	*Salix maccalliana*	ESDMV	-
Pseudomontic willow	*Salix pseudomonticola*	ESDMV	-

Note :ESDMV : Species likely to be designated threatened or vulnerable; - : no status

20.3.3 Geochemical Characterization of Waste Rock, Ore, and Tailings

Characterization of Waste Rock

Former static tests, done for the previous mine, showed that most of the waste rock are potentially acid generating (PAG) under the condition of the test. Although, millions of waste rock have been weathered on the mine site for more than two decades, contact water is still neutral. It is suspected that the potential of neutralisation is not only attributable to carbonates as considered in static tests but are provided by other mineral like silicates. An intensive characterization project is on going with kinetic tests in laboratory, on the field, and at different scales to identify any neutralizing minerals and measure the lag time before the onset of acid rock drainage.

Five main lithological units are recognized in the Troilus area:

1) mafic to felsic volcanic sequence
2) diorite (metadiorite) and brecciated diorite
3) cross-cutting felsic dikes
4) mafic to ultramafic intrusive
5) younger, post-deformation granitic intrusions crosscutting these other rock units

The ongoing research project focusses on rocks from J4 but will be extended to 87 and South-West. Based on geology and mineralogy, the two primary rock units at J4 are (1) volcanics and (2) diorite.

Therefore, samples of these two rock units are tested with various types of geochemical static and kinetic tests to understand better their potentials for acid rock drainage (ARD) and metal leaching (ML). Ultimately, the research objective is to determine how J4 waste rock could affect water quality in the short and long terms.

The research project involves static tests (acid base accounting (ABA), neutralization potential (NP), metal content, whole rock, etc.) and kinetic tests:

- eleven 1-m high columns placed on the mine site and contact water collected every month
- two 3-m high columns placed on the mine site and contact water collected every month
- 4 humidity cells test in laboratory

Table 20-2 and Table 20-3 are the description of rock units in each of the field columns.

Table 20-2: Description of the Eleven 1-m Columns

Column #	Description
F1	low-sulphur non-brecciated diorite (Unit I2J)
F2	mean-sulphur non-brecciated diorite (Unit I2J)
F3	high-sulphur non-brecciated diorite (Unit I2J)
F4	low-sulphur brecciated diorite (Unit I2J; BR)
F5	mean-sulphur brecciated diorite (Unit I2J; BR)
F6	high-sulphur brecciated diorite (Unit I2J; BR)
F7	mean-sulphur undifferentiated volcanics (Unit V)
F8	mean-sulphur felsic intrusives (Unit I1)
F9	composite J4 ore (composite Ore units)
F10	mean-sulphur non-brecciated diorite (Unit I2J)
F11	mean-sulphur brecciated diorite (Unit I2J; BR)

Table 20-3- Description of the Two 3-m Columns Filled with Existing Waste Rock from J4

Column #	Description
E1	heavily oxidized J4 waste rock
E2	relatively unoxidized J4 waste rock

The columns are sampled every month since October 2021 and water quality is analyzed. It is expected that the contact water will reach its maximum equilibrium concentration within several weeks or a few months. Those columns should be monitored for at least 3 years. Figure 20-7 shows the columns built at the site.

Figure 20-7: Field Columns Built on Site

Source: Troilus Gold

For waste rock, the many integrated studies included: (1) on-site monitoring of full-scale water chemistry; (2) on-site kinetic tests holding up to about 300 kg of various rock units; (3) dozens of carefully selected core intervals based on sulphur geostatistics, dominant rock units, and 3D spatial distributions; (4) several types of laboratory kinetic tests; multi-faceted mineralogy using x-ray diffraction, scanning electron microscopy with energy dispersive x-ray spectroscopy, visual petrographics, and microscopic Raman spectroscopy; and (5) compilations of silicate-mineral stoichiometry and reaction rates in various pH ranges. Those studies demonstrated the neutralization potential (NP) of silicates not taken into account for the standard ABA accounting. The conclusion from the report Prediction of ARD Potential in the J4, 87, and Southwest Ore Zones -Phase 1: Based on Generic ARD Criteria is that the majority of the rock that will be mined is non-PAG. Work is continuing to integrate this information with the mining sequence.

Characterization of Ore and Tailings:

Metallurgical pilot plant tests done for the FS study produced five samples of tailings for the geochemical characterization. Static tests have been performed on these samples and show that some samples could be considered PAG under the Québec Regulation according to the Guide de caractérisation des résidus miniers et du minerai Juin 2020, but none would be PAG according to the Prediction Manual for Drainage Chemistry from Sulphidic Geologic Materials (MEND, 2009).

It is important to consider in light of these geochemical results:

- there is already a tailings pond on the site which does not generate acid or metals

- the mineralized zones to be exploited and the metallurgical processes to be used will be the same as in the previous operation
- future mine tailings will have a chemical and mineralogical composition similar to that of the tailings already on the site
- it is expected that the tailings will not generate AMD or NCD since the current tailings facility has demonstrated these findings

20.3.4 Social Context

The Project is within the Eeyou Istchee Territory of the Mistissini Cree First Nation, and on the traditional trapping territories of the tallymen who live on the territory. In June of 2018, the Mistissini Cree First Nation and Troilus Gold signed a Pre-Development Agreement (PDA), which outlines the protocol for working with the Mistissini Cree through the exploration program and defines the steps towards developing an Impact Benefit Agreement (IBA) that is mutually beneficial to both entities to move into the development and production phases of the project.

Troilus keeps good relations and has frequent exchange sessions with the Cree Nation of the Eeyou-Istchee James Bay Region, and in particular the Cree Nation of Mistissini, the First Nations community whose traditional land use and economic activities may be most directly impacted by the company's development. Troilus maintains a community liaison office in Mistissini and communicates regularly with impacted families, the Chief and Council in Mistissini and other community organizations such as the Cree Mineral Board, the Cree Trappers Association, and the Board of Education to keep the community apprised of developments. In August 2020, the Company became the first mineral exploration company to obtain the UL 2723: ECOLOGO Certification Program for Mineral Exploration Companies. The Québec Mineral Exploration Association launched the standard in November 2019 to recognize and promote environmental, social, and economic best practices: the first certification of its kind for mineral exploration companies which enables companies to communicate their commitment to the environment, human health, well-being of the community, and fair economic practices to both investors and stakeholders. The standard is administered by Underwriters Laboratories, an independent, safety testing, certification and inspection organization accredited by the Standards Council of Canada, with a trusted name in third party testing and certification for more than 125 years.

Troilus provides support to community building events and activities in Mistissini, Chibougamau and Chapais which have included over the past year sponsorship of hockey tournaments, fishing derbies, curling bonspiels, art exhibitions and the annual United Way golf tournament.

20.3.5 Socio-Demography

In 2021, the population of Mistissini was 3,731 inhabitants, while it was 3,523 inhabitants in 2016 (Statistics Canada, 2022a). This represents a population increase of 5.9%. The population density per square kilometre is 4.6. In 2016, the average age of the population was 29.8 years (29.0 years for men and 30.5 years for women) while the median age was 26.5 years (25.4 years for men and 27.7 years for women) (Statistics Canada, 2017a). The average size of private households was 3.9 people in 2016. A total number of 670 families include a couple while 205 families are single parents. The first official language spoken is English, for both men and women, while the language most spoken at home is Cree.

Of a total of 2,440 people, 1,325 have no certificate, diploma, or degree, 205 have a high school diploma or equivalency certificate and 910 have a post-secondary certificate, diploma, or degree.

In 2021, the population of Chibougamau was 7,233 inhabitants, while it was 7,504 inhabitants in 2016 (Statistics Canada, 2022b). This represents a population decline of 3.6%. The population density per square kilometre is 10.4. In 2016, the average age of the population was 39.5 years (39.2 years for men and 39.8 years for women) while the median age was 39.8 years (39.4 years for men and 40.2 years for women) (Statistics Canada, 2017b). The average size of private households was 2.3 people in 2016. A total of 1,890 families include a couple, while 325 families are single parents. The first official language spoken is French, for both men and women. The most spoken language at home is also French. Of a total of 6,025 people, 1,535 have no certificate, diploma, or degree, 1,090 have a high school diploma or equivalency certificate and 3,395 have a post-secondary certificate, diploma, or degree.

In 2021, the population of Chapais was 1,468 inhabitants, while it was 1,499 inhabitants in 2016 (Statistics Canada, 2022c). This represents a population decline of 2.1%. The population density per square kilometre is 23.6. In 2016, the average age of the population was 41.4 years (41.6 years for men and 41.1 years for women) while the median age was 43.8 years (44.2 years for men and 43.5 years for women) (Statistics Canada, 2017c). The average size of private households was 2.2 people in 2016. A total of 400 families include a couple while 55 families are single parents. The first official language spoken is French, for both men and women. The most spoken language at home is also French. Of a total of 1,215 people, 405 have no certificate, diploma, or degree, 185 have a high school diploma or equivalency certificate and 625 have a post-secondary certificate, diploma, or degree.

20.4 Stakeholder Engagement

20.4.1 Consultation and Information Activities

Since the purchase of the Site in 2017, several discussions and consultations have taken place with the Cree community of Mistissini, which was closely involved in the former mining operation. These were mainly related to obligations related to site closure and environmental monitoring.

In addition, a pre-development agreement was signed with the Cree community of Mistissini in October 2017 and amended in July 2018 regarding the development of the Troilus mining project.

The following authorities and organizations were met in December 2021 by representatives of Troilus Gold as part of the consultation activities carried out for the Troilus mining project:

- James Bay Regional Administration
- Chapais economic development
- Chibougamau Economic Development
- City of Chapais
- City of Chibougamau

These first consultation activities aimed to establish a dialogue with the stakeholders. They provided an opportunity to present the outline of the project at its current stage and to collect initial comments and concerns. The main issues and comments raised during this first activity are briefly presented in section 3.2. Other meetings are to be expected and the plan for future mobilization is described in section 3.3.

Since October 2019, a monthly report of the activities that are taking place on the site or that are to come is sent to the environmental administrator of the Cree Nation of Mistissini and to the families affected (Awashish, Petawabano and Neeposh).

Troilus Gold participated in several information and consultation meetings with members of the Cree community of Mistissini, families whose trapping territory overlaps the project site (M-34A, M-39 and M-40) as well as other stakeholders.

In October 2021, Troilus Gold sent an invitation to the authorities and organizations below to know their interest in participating in the consultations related to the development of the project:

- Cree Trappers Association of Mistissini
- Council of Mistissini Elders
- Mistissini Youth Council
- Nibischii Corporation
- Grand Council of the Crees
- Cree Nation of Mistissini

All these stakeholders have expressed their interest in participating in these consultations.

20.4.2 Main Issues and Concerns

The main issues and comments raised by the authorities and organizations that met are summarized in Table 20-4 below.

Table 20-4: Main Issues and Comments Raised to date

Subject	Main Issues and Comments
Socia-Economic Aspects	Labor Shortage Housing needs Local mine employment rate/number of job Work schedule Retention of workers in the region (limit fly-in/fly-out) Local and regional economic benefits
Fauna	Protection of wildlife species with a precarious status Fish and fish habitat protection
Air Quality	Dust emission from the mining site
Water Quality	Preservation of surface water quality
Other Social Aspects	Ground transport Waste management
Consultation	Periodic information and consultation of stakeholders Equity between the efforts deployed to Aboriginal communities and non-Aboriginal communities Duplication of Federal and Provincial process

The main issues and comments raised so far by the Indigenous authorities and organizations met are presented in Table 20-5.

Table 20-5: Main Issues and Comments Raised to date by Indigenous Authorities and Organizations

Subject	Main Issues and Comments
Environment	Dust emission, particularly coming from the tailings area Creek diversion, possible flooding Risk of oil spill Waste rock runoff collection
Fauna	Movement of animals in the area
Light Pollution	Dark Sky Preserve Project
Cultural Aspects	Increased safety of traditional activities in restored sectors (development design) Circulation of land users in the area Distribution of economic benefits in the community Priority for impacted families' members for training and jobs
Social-Economic Aspect	Impact on touristic activities Taxation of income according to employment status Recognition of training and experience, including for women
Other Social Aspects	Difficulty of long work schedules for family life, particularly for women. Ground transport (road condition) Ground transport (user safety)

20.4.3 Engagement Plan

The stakeholders mentioned previously will be part of the future engagement plan.

The future mobilization plan will notably include the following main elements:

- the distribution of a bi-annual newsletter
- the establishment of information and consultation sessions regularly according to the progress of the project and the requests of the stakeholders
- sending reminder emails and information links regarding the important dates of the main project activities
- if necessary, planning working meetings on specific subjects with the stakeholders concerned

Troilus Gold recognizes the positive impact that the former Troilus mining operation had on relations with the Cree community of Mistissini. In this context, Troilus Gold wishes to pursue an inclusive and transparent approach and has therefore committed to taking the following main steps:

- continue to hold information and mobilization meetings with members of the Cree community of Mistissini frequently face-to-face and virtually
- continue to have individual meetings with the tallymen on the territory of the Troilus mining project

- continue distribution of a bi-annual community newsletter
- if necessary, make visits to the project site with members of the Cree community of Mistissini
- maintain the employment of a Cree liaison officer to facilitate communications with members of the Cree community of Mistissini
- creation of an exchange table with representatives of various sectors of interest in the Mistissini community (hunters, young people, elders, etc.)
- preparation of an impact and benefit agreement
- gathering information from Indigenous experts on Indigenous knowledge

20.5 Closure Plan

The closure plan will be developed with the required level of detail to be filed with the Provincial Impact Study; this would replace the current closure plan that is currently in effect. The new closure plan will include consultations with community and land users for the future use of the land. People and wildlife needs will be taken into consideration in the final closure design.

Items in the mandatory closure plan include:

- a description of the closure activities (dismantling of infrastructures, revegetation, monitoring, etc.)
- a financial guarantee for 100% of the closure costs, including some contingency

The main focus for the closure plan is long term water quality prediction, to be able to put in place appropriate protection if required.

As the project advances through various stages of study, the Closure Plan will have to be updated in the future Project incorporating updated practices and regulations The closure plan will be revised regularly as the Project enters the operation phase and with more scrutiny when getting closer to the closure phase.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Cost

21.1.1 Summary

This section outlines the capital costs for the 50 kt/d Troilus Gold Copper Project. All costs unless otherwise noted are in United States currency (US$).

The capital cost estimate (estimate) includes all the direct and indirect costs along with the appropriate estimating contingencies for all the facilities required to bring the Project into production, as defined by this Study. All equipment and material are assumed to be new. Labour costs based on the statutory laws governing benefits to workers in effect in Québec at the time of the estimate. The estimate does not include any allowances for scope changes, escalation, and exchange rate fluctuations. The execution strategy is based on an engineering, procurement, and construction management (EPCM) implementation approach.

The capital cost estimate for the Project was developed to provide an estimate suitable for a Feasibility Study phase including cost to design, construct, and commission the facilities. The estimate produced is described as a Class 3 with an expected accuracy of +15% -10%. This classification is based on the AACE international standard.

Table 21-1 shows the Functional Area summary of total capital costs for the 50 kt/d Project including mine, process plant, TSF on-site infrastructure and Project in-directs for the major areas. The initial, sustaining, and total capital costs for the Project is estimated to be $1,351 million dollars expressed in H2 2023 USD price levels exclusive of duties and taxes.

Table 21-1: Troilus Project Capital Cost Estimate

Area	Initial Capital (M$)	Sustaining Capital (M$)	Total Capital (M$)
Open Pit – Prestrip (capitalized)	213.0	-	213.0
Open Pit - Capital	45.3	99.3	144.6
Open Pit Mining - Subtotal	258.3	99.3	357.6
Processing	443.0	15.1	458.1
Infrastructure	100.3	27.7	128.0
Environmental	10.7	67.4	78.2
Indirects	173.0	50.5	223.4
Contingency	89.3	16.6	105.9
Total	**1,074.6**	**276.6**	**1,351.2**

The capital cost estimate expressed in the work breakdown structure is shown in Table 21-2 below. It should be noted that the environmental bonding is outside of the structure and is shown as a line item below the total for completeness.

Table 21-2: Troilus Level 1 Project Capital Cost Estimate

Area	Initial Capital (M$)	Sustaining Capital (M$)	Total Capital (M$)
000 Construction Distributables	93.5	10.1	103.5
100 Treatment Plant Costs	400.7	8.0	408.7
200 Reagents & Plant Services	41.9	7.1	49.0
300 Infrastructure	100.3	27.7	128.0
400 Mining	258.3	99.3	357.6
500 Management Costs	47.0	7.0	54.0
600 Owners Project Costs	32.9	24.8	57.7
Contingency	89.3	16.6	105.9
Subtotal	**1,063.9**	**200.6**	**1,264.5**
Environmental (Bonding, Closure reclamation)	10.7	76.0	86.7
Total	**1,074.6**	**276.6**	**1,351.2**

21.1.2 Estimate Responsibility

This capital cost estimate reflects the joint efforts of Lycopodium, Troilus and specialty consultants retained by Troilus – AGP and Golder. Lycopodium was responsible for compiling the submitted data into the overall estimate but did not review or validate the inputs from Troilus or its other consultants. Table 21-3 outlines the responsibilities of each company for input of information into the capital cost estimate. Each consultant provided input to the capital cost estimate appropriate to a +15% -10% accuracy estimate, including all related In-direct costs and allowances.

Table 21-3: Capital Cost Estimate Responsibilities

Company	Responsibility
Lycopodium	Process plant, on-site infrastructure, sustaining cost for Process plant warehouse, spares (operating and critical) , gold room and gravity concentrator, and purchase of construction camp only.
AGP	Mining
WSP	Tailings thickening, tailings pipeline, TSF, reclaim water pipeline and site water management
Troilus	Owner’s costs, closure costs, salvage values and taxes (included in the financial model)

21.1.3 Work Breakdown Structure

The Work Breakdown Structure (WBS) for the Project was developed by a Lycopodium and included four WBS organization levels and a series of discipline codes.

The capital cost for the Level 2 WBS is shown in Table 21-4.

Table 21-4: Level 2 Initial Capital Estimate Summary by Major Area

Main Area	Sub Area	Subtotal Cost (M$)	Contingency Cost (M$)	Total Cost (M$)
000 Construction Distributables	010 Construction Distributables - Contractors	52.9	10.6	63.5
	020 Site Construction Distributables General	1.6	0.3	2.0
	050 Construction Operations	6.1	1.2	7.3
	060 Construction Accommodation	32.8	6.6	39.4
000 Construction Distributables Total		**93.5**	**18.7**	**112.2**
100 Treatment Plant Costs	110 Treatment Plant - General	14.5	1.9	16.5
	120 Feed Preparation	77.9	8.2	86.1
	130 Milling	224.2	23.8	248.0
	150 Flotation & Concentrate Handling	69.6	7.7	77.2
	170 Gold Room	-	-	-
	180 Tails Handling	14.5	1.6	16.1
100 Treatment Plant Costs Total		**400.7**	**43.1**	**443.8**
200 Reagents & Plant Services	210 Reagents	8.6	1.0	9.6
	220 Water Services	6.4	0.9	7.3
	230 Plant Services	4.2	0.5	4.7
	240 Air Services	1.3	0.2	1.5
	250 Fuels	0.0	0.0	0.0
	260 Electrical Services	21.3	2.1	23.5
200 Reagents & Plant Services Total		**41.9**	**4.7**	**46.6**
300 Infrastructure	310 Infrastructure - General	0.7	0.1	0.8
	320 Environmental	61.8	8.3	70.1
	330 Water & Sewerage	2.4	0.2	2.6
	340 Power Supply	10.6	1.1	11.7
	350 Tailings Dam	15.3	1.7	17.0
	360 Buildings - Admin & Security	7.1	0.7	7.8
	370 Buildings - Plant	0.2	0.0	0.3
	380 Permanent Accommodation	2.2	0.3	2.4
300 Infrastructure Total		**100.3**	**12.4**	**112.7**
400 Mining	410 Mining-General	1.2	0.1	1.3
	420 Mine Establishment	242.0	1.5	243.5
	450 Mining Facilities	15.1	1.6	16.7
400 Mining Total		**258.3**	**3.2**	**261.5**
500 Management Costs	510 EPCM - Home Office	11.8	1.2	12.9
	520 EPCM - Site	32.2	3.3	35.5
	550 Vendor Representatives	3.1	0.6	3.7
500 Management Costs Total		**47.0**	**5.1**	**52.1**
600 Owners Project Costs	610 Owners Costs - General	22.9	1.3	24.2
	620 Plant & Admin Pre-Production	9.4	0.8	10.1
	640 Spare Parts	0.2	0.0	0.2
	670 Plant Mobile Equipment	0.4	0.0	0.5
600 Owners Project Costs Total		**32.9**	**2.1**	**35.0**
	Subtotal	**974.6**	**89.3**	**1,063.9**
Environmental		10.7	-	10.7
	Total	**985.3**	**89.3**	**1,074.6**

The exchange rates used in the compilation of the estimate are noted in Table 21-5.

Table 21-5: Currency Exchange Rate

Currency	USD
CAD	0.74
USD	1.00
Euro	1.10
AUD	0.67

21.1.4 Mine Capital Costs

Open Pit Mining

The mining equipment capital costs were developed with the use of financing of the fleet. Base capital costs were obtained and developed with options, then finance parameters are applied. In conversation with the various vendors the down payment varied between 10% and 20% depending on equipment and vendor. For the FS, a downpayment of 10% was applied. The remaining finance costs are then distributed over the operating costs discussed later in Section 21.2.2.

Equipment pricing was based on quotations from local vendors predominantly with some smaller equipment information from AGP's database of recent projects. The base costs provided by the vendors are included in the calculation for each unit cost and options were added to that as shown in Table 21-6 with the full finance cost.

Table 21-6: Major Mine Equipment – Capital Cost and Full Finance Cost (US$D)

Equipment	Unit	Capacity	Capital Cost	Full Finance Cost
Production Drill	mm	165	1,242,200	1,532,900
Production Drill	mm	203	3,427,400	4,229,400
Production Loader	m^3	23	7,692,600	10,260,300
Hydraulic Shovel	m^3	34	14,059,300	17,349,100
Hydraulic Excavator	m^3	6.5	1,220,700	1,658,700
Haulage Truck	t	229	5,125,900	6,325,400
Haulage Truck	t	91	1,733,300	2,311,900
Haulage Truck	t	40	703,700	938,600
Crusher Loader	m^3	11.5	3,248,100	4,332,400
Track Dozer	kW	565	2,081,500	2,568,500
Grader	kW	163	1,346,700	1,829,800

Some items such as spare truck boxes and spare shovel buckets were capitalized and purchased at the same time as the mine equipment. In the case of the haulage trucks, spare boxes are estimated one spare box will be required for every four trucks. For the hydraulic shovels and loader, the estimate is that one spare bucket per two loading units will be required.

The distribution of the capital cost is completed using the units required within a period. If new or replacement units are needed, that number of units, by unit cost, determines the capital cost for that period. There is no allowance for escalation in any of these costs. Timing of major capital equipment costs is one year in advance of the need for that piece of equipment. Therefore, if the equipment is required in Year 1, the cost is charged in Year -1. The finance calculation adjusts that, so the cost of the financing is in the year the equipment arrives on site. The finance cost less the downpayment is an operating cost.

The number of units are determined by the mine schedule and the operating cost estimate for required operating hours. These were balanced over periods of time so if there are fluctuations in the hours from period to period, or year to year, they are distributed for the entire equipment fleet to balance the hours.

Replacement times for the equipment are average values from AGP's experience. Options around rebuilds and recertification of equipment like track dozers is not considered, nor is used equipment, although that should be considered during the purchase of the mine fleet.

The balancing of equipment units based on operating hours is completed for each major piece of mine equipment. The smaller equipment was based on number of units required, based on operational experience. This includes such things as pickup trucks (dependent on the field crews), lighting plants, mechanics trucks, etc.

The most significant piece of major mine equipment is the haulage trucks. At the peak of mining in Year 14 the haulage fleet is 44 units of 229 t size which are necessary to maintain mine production. The maximum hours per truck/per year are set at 6,000. There are periods where the maximum hours per unit are below 6,000. In those cases, the hours required are distributed evenly across the number of trucks within the fleet.

The other major mine equipment is determined in the same manner. Therefore, in some instances the smaller production loaders have a longer period of life (same number of hours between replacements) due to the sharing of hours with the other units in the fleet.

The support equipment is usually replaced on a number of year's basis. For example, pickup trucks are replaced every four years, with the older units possibly being passed down to other departments on the mine site, but for capital cost estimating new units are considered for mine operations, engineering, and geology.

The timing of equipment purchases, initial and sustaining, are shown in Table 21-7. If the Project were to advance without financing, the quantity of units would remain the same. The forecast operating life by unit is also shown in the table. Table 21-8 shows the total number of units on site by year.

Table 21-7: Equipment Purchases – Initial and Sustaining

Equipment	Unit Life (hrs.)	Yr - 2	Yr - 1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14-21
Drill (165 mm)	25,000	2		1				1	1				1	1			4
Drill (203 mm)	35,000		5	1	1				2	4	1				1	6	1
Loader (23 m^3)	50,000	1	1														
Hydraulic Shovel (34 m^3)	75,000		2	1	1											1	2
Hydraulic Excav (6.5 m^3)	30,000	2								1	1						2
Truck (229t)	70,000	5	13	8	2	2	4	4	3					5	11	13	8
Truck (91t)	50,000	3															
Truck (40t)	20,000	3															
Crusher Loader	35,000			1								1					1
Tracked Dozer	35,000	4	2							1	5						6
Grader	25,000	2	1							1	2						3

Table 21-8: Equipment Fleet Size

Equipment	Unit Life (hrs.)	Yr - 2	Yr - 1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14-21
Drill (165 mm)	25,000	2	2	3	3	3	3	3	2	2	2	2	2	2	2	2	4
Drill (203 mm)	35,000	-	5	6	7	7	7	7	7	7	7	7	7	7	7	7	8
Loader (23 m^3)	50,000	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Hydraulic Shovel (34 m^3)	75,000	-	2	3	4	4	4	4	4	4	4	4	4	4	4	3	3
Hydraulic Excav (6.5 m^3)	30,000	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Truck (229t)	70,000	5	18	26	28	30	34	38	41	41	41	41	41	36	28	36	44
Truck (91t)	50,000	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Truck (40t)	20,000	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Crusher Loader	35,000	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Tracked Dozer	35,000	4	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Grader	20,000	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3

The portion of the mining capital that is not financed is tabulated in Table 21-9.

Table 21-9: Mining Capital Cost Estimate (US$D)

Equipment	Preproduction Year -2, -1	Sustaining	Total
Pre-Production Stripping (420)			
Pre-Production Stripping	212,996,200	-	212,996,200
Mining Equipment (420)			
Mining Equipment	25,434,400	67,740,000	93,174,400
Miscellaneous Mine Capital (420)			
Engineering Office Equipment	1,111,100	-	1,111,100
Dispatch System	833,300	277,800	1,111,100
Pit Communications	297,600	-	297,600
Dewatering System	1,060,300	20,182,200	21,242,500
Pit Area – Clear/Grub	296,300	-	296,300
Subtotal – Miscellaneous Mine Capital	**3,598,600**	**20,460,000**	**24,050,600**
Mine Infrastructure (410,450)			
Explosive Plant Pad (410)	1,194,000	-	1,194,000
Mine Workshop (450)	15,121,700	11,062,000	26,183,700
Subtotal – Mine Infrastructure	**16,315,700**	**11,062,000**	**27,377,700**
Total Mine Capital	**258,344,900**	**99,262,000**	**357,606,900**

Pre-Production Stripping

The mine is scheduled to initiate mining in Year -2. The material moved will be used to develop the mine roads to the SW pit, the stockpile platform, initiate the buttress for the tailings area, build the diversion dyke and provide mill feed for the stockpile. A total of 63.9 Mt will be mined in this time period and the costs are being attributed to capital. This is expected to cost $213.0 M. This cost includes all associated management, dewatering, drilling, and blasting, loading, hauling, support, engineering and geology department labour, and grade control costs.

Mining Equipment

The initial downpayments of the mining equipment are included here. As well, various pieces of mining equipment were not financed including items such as spare truck boxes, or shovel buckets. It also includes the blasting truck, and pump truck, as well as the ambulance, fire truck and associated rescue equipment, a 35-t rough terrain crane and a 100-t lowboy and tractor for moving drills and dozers between the various pit areas.

Miscellaneous Mine Capital

This category covers the engineering office equipment which includes mining software, computers, plotters, GPS equipment, etc. It also covers the initial dispatch office system. Actual dispatch items for the units are included with the mine trucks and shovels cost estimate.

The dewatering system is also included in this category. It includes all necessary pumps, pipes, and cabling to keep the pits dry for mining operations.

Mining Infrastructure

Two pieces of mine infrastructure were separated in the costing. These are the concrete pad for the explosives plant and the truck workshop. These costs were prepared by Lycopodium as part of the overall site infrastructure.

21.1.5 Process Plant and Infrastructure Capital Cost

Estimating Methodology

Engineering lists, layout drawings, process flow diagrams, other process deliverables and a 3D model have been produced with sufficient detail to permit the assessment of the engineering quantities for concrete, steelwork, mechanical, and electrical for the process plant and associated infrastructure. Historical database for similar projects was also available to complement information where required.

The unit rates and labour rates are based on historical rates and budget quotes. Budget quotes for mechanical and electrical equipment were obtained from reputable international suppliers.

Quantity Development

The Project works were quantified to represent the defined scope of work and to enable the application of rates to determine costs.

Quantity information was derived from a combination of sources and categorized to reflect the maturity of design information as follows:

- **Material Take-Offs**: detailed quantities derived from drawings and Troilus 3D Model and General Arrangement drawings
- **Factored**: quantities derived from percentages applied as a factor based on experience or similar projects
- **Allowance:** lump sum added to the cost estimate to account for costs that are certain to occur, but they cannot be identified with any accuracy

The derivation of quantities is provided in Table 21-10.

Table 21-10: Derivation of Quantities

Classification	MTO Prepared	Factored
Site works	✓	
Concrete + Detailed Excavation	✓	
Structural Steel	✓	
Platework	✓	
Mechanical Equipment	✓	
Process Piping		✓
Electrical Bulks	✓	
Electrical Equipment	✓	
Instrumentation and Control	✓	

It should be noted the typical design growth applied to bulk quantities was removed per Troilus' instructions.

Pricing Basis

Table 21-11 identifies the sourcing of costs included in the estimate as described further in detail in Section 21.1.6. The bulks are all escalated costs from the quotes received in 2021 and early 2022.

Table 21-11: Supply Cost Source

Classification	Total Supply Cost $	Allowance / Factored	Estimated / Historical Pricing	Budget Pricing
B Earthworks	29,799,987	0.3%	88%	12%
C Concrete	40,689,764	3%	1%	96%
D Steelwork	17,610,312	12%	19%	69%
E Platework	6,554,832	1%	23%	76%
E Tankage	2,345,216	-	86%	14%
F Mechanical	154,293,614	1%	2%	97%
G Piping	18,425,132	74%	17%	9%
H Electrical	52,963,917	-	42%	58%
J Instrumentation & Control	2,494,515	26%	-	74%
M Buildings & Architectural	37,941,834	20%	13%	68%

21.1.6 Quantity and Pricing Basis by Discipline

Earthworks

Material take-offs were prepared for the earthwork quantities based on the recommendations made by the WSP. A large portion of the earthworks will be self-performed by Troilus using local and existing material on site. The cost for this work has been estimated in conjunction with AGP. The rates for the process plant earthworks were based on a budget quote provided by a local contractor. Historical rates were used to price the drains and CSP culverts. The specialized earthworks related to the WSP scope has been based on historical rates.

Concrete

Material take-offs have been prepared for the estimate.

Rates for concrete works were based on a budget quote and have been escalated to align with base date of estimate. Allowances were added for the laydown area and reclaim stockpile tunnel steel sections. Allowances for concrete heating and hoarding were included separated in the estimate based on historical data adjusted to Troilus construction period.

Rates and quantities were prepared on a composite per cubic metre basis for each specific type of concrete assembly.

Steelwork

Structural steel quantities were estimated using material take-off for each process plant area.

Structural steel installation and supply is based on a recent budget quote for a similar project. The steel supply cost is from Southeast Asia.

Platework and Shop/Field Fabricated Tanks

Platework and tankage quantities were provided in the mechanical equipment list with estimated weights.

Rates for the supply and fabrication were based on historical rates. The supply rate for the platework is based on Southeast Asian Supply.

Mechanical Equipment

Major mechanical equipment has been priced based on budget quotes with remaining equipment based on historical rates. Budget quotes were received in 2023 for the majority of the equipment packages. A discount amount has been included in estimate assuming that a bundle pricing strategy will be adopted by the project procurement.

Mechanical installation hours and rates for major equipment are based on contractor budget quote. A productivity factor (PF) of 1.2 was only applied for miscellaneous items if the hours were estimated from the historical data and had similar working conditions to Troilus.

Piping

Material take-offs and pricing were developed for Site Water Management Piping by WSP. The surface overland piping was quantified by Lycopodium. The process plant piping bulks were factored based on similar historical information identified for each plant area.

WSP estimated the supply and install their scope of work, their rates were benchmarked against database rates. The overland piping supply and install is based on 2023 historical rates.

Electrical/Instrumentation

Major electrical equipment and electrical bulks in the Lycopodium scope were quantified. The electrical equipment pricing is based on updated 2023 budget quotes and the remaining supply prices are based on historical rates.

Instrumentation Control System, which includes PLC hardware and software, is based on a budget quote. Instrumentation bulks were quantified, and a mix of budget and historical pricing was used to price the supply cost. Installation hours were quoted by contractors.

The substation upgrade is based on the Stantec estimate completed in 2022 and has been escalated. The transformer in the substation upgrade is based on a budget quote.

Buildings

The supply and installation cost for the Pre-engineered buildings were based on budget quotes. Building Fit Out and HVAC allowances are included and has been estimated based on the volume of the buildings and historical fit out costs.

Mobile Equipment List

The new process mobile equipment for this Project will be leased, a 5% down payment for this equipment has been included in the estimate.

Contractor Indirects

Contractor indirects are based on historical cost and budget quotes and include offsite management, onsite staff and supervision above trade level, crane drivers, equipment and labour mobilization and demobilization. The contractor indirect ratio for concrete and earthworks are based on the budget quotes and SMPEI is based on historical ratios.

Construction indirect costs for all direct labour is included in the capital cost estimate. This is inclusive of PPE, travel, and clothing. Fuel has also been included in the contractor indirects and has been calculated based on the package hours. Contractor's temporary facilities have been included.

21.1.7 Indirects

Engineering, Procurement, and Construction Management (EPCM)

The EPCM cost was bottoms up estimated by Lycopodium for their scope of work based on 36 months EPCM duration and 3 months of commissioning. WSP provided an estimate EPCM cost for their scope of work.

The EP scope of services for the Project will include basic engineering, detailed engineering, procurement services for equipment and materials purchases, project management and controls. The CM scope includes for site construction management and quality assurance.

Construction Indirects

Construction indirect costs were estimated based on construction duration and includes for items not within contractor's scope of supply. This includes temporary facilities, warehousing, surveying, environmental services, third party testing, health services and safety. The estimate accounts for the infrastructure already available at site as directed by Troilus. Fuel, meals and accommodation, vehicles, have been estimated.

The construction camp leasing cost is based on a budget quote. Construction room and board has been estimated based on camp loading, construction duration and recent pricing for Canadian camp maintenance cost and is included in the capital cost estimate as a direct cost. Bussing to and from site has been estimated based on the camp loading and estimated bussing costs.

Spares

Commissioning spares for major equipment were quoted by vendors. The cost of the commissioning spares for the balance of the equipment was factored. The capital and operating spares are included in the sustaining cost estimate.

Vendor Representatives

Some equipment will require vendor representation during construction and/or commissioning.

A provision has been included in the estimate to cover the vendor representatives' services, based on major mechanical equipment packages.

Freight

Freight was based on a percentage of the supply cost. These factors were obtained from vendor quotations, and if they were not provided, an approximation of 8% to 20% of equipment supply cost was included based on historical rates depending on sourcing of materials and equipment.

Owner's Costs

The owner's cost has been estimated and included in the estimate. The owner's cost includes for the oversight management team and other expenses as directed by Troilus.

Contingency

An amount of contingency has been provided in the estimate to cover unknown variances between the specific items allowed in the estimate and the final total installed Project cost. The contingency does not cover scope changes, design growth, etc., or the listed qualifications and exclusions.

Contingency has been applied to the estimate on a line-by-line basis as a deterministic allowance by assessing the level of confidence in each of the defining inputs to the item cost, these being engineering, estimate basis and vendor or contractor information, and then applying an appropriate weighting to each of the three inputs. It should be noted that contingency is not a function of the specified estimate accuracy and should be measured against the Project total cost that includes contingency. The resultant contingency for the Capital Cost Estimate is approximately 9% of total Project cost.

Escalation

There is no allowance for escalation beyond Q4 2023 in the estimate.

21.1.8 Qualifications and Assumptions

The capital cost estimate is qualified by the assumptions listed in Table 21-12.

Table 21-12: Qualifications and Assumptions

Qualifications / Assumptions
GENERAL
Mobile equipment will be leased.
The base date for the bulk of pricing estimate is first quarter 2023 (4Q 2023).
Some labour rates, materials and equipment supply costs are escalated from fourth quarter 2022 to fourth quarter 2023, while other labour rates were based on Q4 2023 quotes.
Prices of materials and equipment with an imported content have been entered in native currencies and converted to USD at the rates of exchange stated previously in this document.
Troilus will provide sufficient power and water for construction.
No allowance has been considered for civil unrest or disturbances.
Troilus will secure and provide all required permits.
The estimate allows for supply of structural steel and platework from Southeast Asia. Import duties are excluded from the estimate.
Communications network and data for construction facilities to be free issued by the Owner.
Construction insurance included in Owner's Cost.
EARTHWORKS
The bulk earthworks commodity rates that include imported material assume that suitable construction/fill materials will be available from borrow pits within 2 km of the work fronts.
Aggregates will be supplied by AGP, and their unit rate for supply has been carried in the estimate.
CONCRETE
Assume sand, aggregate and cement are by contractors to specification and assumed to be available locally.
The estimate allows for all reinforced bar and mesh for construction to be provided by the concrete contractor.
Mechanical assemblies will be performed in the parallel to the Concrete work in the laydown area. Heavy crane will be used to lift the assemblies in place.
OTHERS
Power supply and switchyard is by Troilus.
INDIRECTS
Accommodation & Meals CM Team – Troilus Camp and meals will be provided to the CM Team
Site transportation from Camps to Site will be provided by Troilus
Travel to Camps or site from the nearest city is assumed be at the Contractor's account, and no payments need to be made for such travel by Troilus

21.1.9 Sustaining Cost Summary

The sustaining capital includes for the following.

- cost from year 1 to 22 for site water drainage and tailings dam
- pumping and pipeline from year 1 to 22 and related operation and maintenance cost
- expansion of mine service building
- water treatment plant
- site water drainage pumping and pipeline operating costs
- process plant warehouse
- spares (operating and critical)

- gold room and gravity concentration
- purchase of construction camp

21.1.10 Exclusions

The following items are specifically excluded from the capital cost estimate:

- permits and licences
- project sunk costs
- escalation beyond the base date of the estimate
- exchange rate variation
- working capital
- scope changes
- further unidentified ground conditions
- extraordinary climatic events
- bonding, permits and legal costs
- national and local taxes and duties
- project insurance cost
- force majeure
- labour disputes
- schedule recovery or acceleration
- cost of financing
- property taxes, corporate and mining taxes, duties
- research and exploration drilling
- salvage values
- no allowances have been made for COVID related costs

21.1.11 Capital Cost Risks

It is foreseen that the cost on many items is currently being impacted due to world/market conditions. This estimate does not take these types of risks into consideration. The capital cost estimate is only valid for Q4 2023.

Given the above-mentioned challenges, capital cost sensitivity should be reviewed as part of the financial model.

21.2 Operating Cost Estimates

21.2.1 Operating Cost Summary

The estimated Project operating costs are shown in Table 21-13 highlighting different periods up to the end of the mine life in Year 22. All costs are reported in US$.

Table 21-13: Troilus Project Operating Costs

Area	Units	Year 1 – 5	Year 6 – 22	Life of Mine (Year 1-22)
Open Pit Mining	$/t moved	2.82	3.01	2.96
	$/t mill feed	13.33	11.09	11.60
Processing	$/t mill feed	5.63	5.64	5.64
G&A	$/t mill feed	1.28	1.09	1.14
Concentrate Trucking, Port, Shipping	$/t mill feed	0.68	0.68	0.68
Total Operating Cost	**$/t mill feed**	**20.92**	**18.51**	**19.06**

General data sources and assumptions used as the basis for estimating the process operating costs include:

- process design criteria in Section 17
- average production rate of 50 kt/d
- manpower requirements as developed by AGP and Lycopodium with input from Troilus
- unit cost of electrical energy of $0.026/kWh
- unit cost of diesel fuel of $1.07/L
- taxes are excluded from the G&A but is applied to the financial model
- refining costs and NSR royalties are excluded from the operating costs but are applied to the financial model
- mobile equipment carried in the process operating cost estimate is assumed to be leased; lease costs are included in the capital cost estimate and mobile equipment costs include fuel and maintenance only

21.2.2 Mine Operating Costs

Mine operating costs are estimated from base principles. Key inputs to the mine costs are fuel and labour. The fuel cost is estimated using local vendor quotations for fuel delivered to site. A value of $1.07/L is used in this estimate.

Open Pit Mine Operating Cost Estimate

Labour cost estimates were based on queries to other operations and recent salary surveys at mines in Québec. Shift schedules are 12-hour shifts with a 4 days on/ 4 days off schedule. Some management positions will be on a 4-days on and 3-days off basis. A burden rate of 35% was applied to all rates. Mine positions and salaries are shown in Table 21-14.

Table 21-14: Open Pit Mine Staffing Requirements and Annual Salaries (Year 5)

Staff Position	Employees	Full Load Annual Salary ($/a)
Mine Maintenance		
Maintenance Superintendent	1	175,000
Maintenance General Foreman	1	144,000
Maintenance Shift Foremen	4	127,000
Maintenance Planner/Contract Admin	2	115,000
Maintenance Clerk	1	70,000
Subtotal	**9**	
Mine Operations		
Mine Ops/Technical Superintendent	1	200,000
Mine Operations General Foreman	1	144,000
Mine Shift Foreman	4	127,000
Junior Shift Foreman	4	116,000
Trainers	2	127,000
Road Crew/Services Foreman	1	127,000
Operations Clerk	1	70,000
Subtotal	**14**	
Mine Engineering		
Chief Engineer	1	165,000
Senior Engineer	1	142,000
Open Pit Planning Engineer	2	130,000
Geotech Engineer	1	130,000
Blasting Engineer	1	115,000
Blasting/Geotech Technician	2	95,000
Dispatch Technician	4	95,000
Surveyor/Mining Technician	2	90,000
Surveyor/Mine Technician Helper	2	80,000
Engineering Clerk	1	70,000
Subtotal	**17**	
Geology		
Chief Geologist	1	165,000
Senior Geologist	1	141,000
Grade Control Geologist/Modeler	2	115,000
Sampling/Geology Technician	4	95,000
Geology Clerk	1	70,000
Subtotal	**9**	
Total Mine Staff	**49**	

The mine staff labour remains consistent for the mine life after the initial recruitment in the pre-production period (Year -2 and -1). This level holds at 49 in the first five years due to trainers in mine operations.

Hourly employee labour force levels in the mine operations and maintenance departments fluctuate with production requirements. A snapshot of the labour makeup for Year 5 is shown in Table 21-15.

Table 21-15: Hourly Manpower Requirements and Annual Salary (Year 5)

Hourly Position	Employees	Full Load Annual Salary ($/a)
Mine General		
General Equipment Operator	16	99,000
Road/Pump Crew	12	96,000
General Mine Laborer	8	92,000
Trainee	4	77,000
Light Duty Mechanic	3	104,000
Tire Technician	6	104,000
Lube Truck Driver	8	94,000
Subtotal	**57**	
Mine Operations		
Driller	40	102,000
Blaster	2	102,000
Blaster’s Helper	4	98,000
Loader Operator	4	102,000
Hydraulic Shovel Operator	16	102,000
Haul Truck Driver	156	95,000
Dozer Operator	16	99,000
Grader Operator	6	99,000
Transfer Loader	3	99,000
Snowplow/Water Truck	11	95,000
Subtotal	**258**	
Mine Maintenance		
Heavy Duty Mechanic	63	104,000
Welder	33	104,000
Electrician	3	104,000
Apprentice	9	96,000
Subtotal	**108**	
Total Hourly	**423**	

Labour costs are based on an owner operated scenario. Troilus is responsible for the maintenance of the equipment with its own employees.

Over seeing all of the mine operations, engineering, and geology functions is a Technical Superintendent. This person would have the Mine Maintenance Superintendent, Mine General Foremen, Chief Engineer, and Chief Geologist reporting to them. The Technical Superintendent would report to the Mine General Manager.

The Mine General Foreman would have the shift foremen report directly to him.

The mine has four mine operations crews, each with a Senior Shift Foremen who has one Junior Shift Foreman reporting to him. For the mine life, there is also a Road Crew/Services Foreman responsible for roads, drainage, and pumping around the mine. This person would also be a backup Senior Mine Shift Foreman. The Mine Operations department has its own administrator.

The Chief Engineer has one Senior Engineer and two open pit engineers reporting to him. The Blasting Engineer is included in the short-range planning group and would double as drill and blast foreman as required. The Geotechnical engineer would cover all aspects of the wall slopes and waste dumps together with two shared technicians in blasting.

The short-range planning group in engineering also has two surveyor/mine technicians and two surveyors/mine helpers. These people will assist in the field with staking, surveying, and sample collection with the geology group; they will have an administrator to assist the team.

In the Geology department, there is one Senior Geologist reporting to the Chief Geologist. There are also two grade control geologists/modellers; one will be in short range and grade control drilling, and the other will be in long range/reserves. There are also four grade control geologists (one per mine operations crew) and one administrator.

The Mine Maintenance Superintendent has the Maintenance General Foreman reporting to him. Eight Mine Maintenance Shift Foremen will report to the Maintenance General Foreman. As well, there are two maintenance planners/contract administrators and an administrator.

The hourly labour force includes positions for the light duty mechanic, and lube truck drivers. Tire changing duties are provided by a contractor. These positions all report to Maintenance. There are generally one of each position per crew. Other general labour includes general mine labourers (two per crew) and trainees (one per crew until Year 10).

The drilling labour force is based on one operator per drill, per crew while operating. This on average is 10 drillers per crew.

Shovel and loader operators peak at 28 in Year 12 and will start to tail off after that. Haulage truck drivers peak at 140 in Year 8 and then taper off to the end of the mine life.

Maintenance factors are used to determine the number of heavy-duty mechanics, welders and electricians are required and are based on the number of drill operators. Heavy duty mechanics work out to 0.25 mechanics required for each drill operator. Welders are 0.25 per drill operator and electricians are 0.05 per drill operator. This method of estimating maintenance requirements is used for each category of the mine operating cost and is summarized in Table 21-16.

Table 21-16: Maintenance Labour Factors (Maintenance per Operator)

Maintenance Job Class	Drilling	Loading	Hauling	Mine Operations Support
Heavy Duty Mechanic	0.25	0.25	0.25	0.25
Welder	0.250	0.25	0.25	0.25
Electrician	0.05	0.01	-	-
Apprentice	-	-	-	0.25

The number of loader, truck, and support equipment operators is estimated using the projected equipment operating hours. The maximum number of employees is four per unit to match the mine crews.

The vendors provided repair and maintenance (R&M) costs for each piece of equipment. These came in the quotations for the capital cost. Fuel consumption rates are also estimated for the conditions expected at Troilus and are used in the detail costs for the mine equipment. The costs for the R&M are expressed in a $/h form.

The various suppliers provided the costs for different tire sizes that will be used during the project. Estimates of the tire life are based on AGP's experience and conversations with mine operators. The operating cost of the tires is expressed in a $/h form. The life of the haulage truck tires is estimated at 4,500 hours per tire with proper rotation from front to back. On the haulage trucks each tire costs $39,800 so the cost per hour for tires is $35.40/h for the truck using six tires in the calculation.

Ground Engaging Tool (GET) costing is estimated from other projects and conversations with personnel at other operations. This is an area of cost that is expected to be fine-tuned during mine operations.

Drill consumables were estimated as a complete drill string using the parts list and component lives provided by the vendor. Drill productivity for the 165 mm drill is estimated at 25.8 m/h and for the 203 mm drill at 24.9 m/h. Equipment costs used in the estimate are shown in Table 21-17.

Table 21-17: Major Equipment Operating Costs – no labour ($/h)

Equipment	Fuel/ Power	Lube/ Oil	Tires	Under-Carriage	Repair & Maintenance	GET/ Consumables	Total
Production Drill (165 mm)	48.33	4.83	-	2.22	88.83	146.26	257.98
Production Drill (203 mm)	11.73	-	-	4.44	69.76	170.91	256.85
Production Loader (23 m^3)	193.33	29.00	69.93	-	139.76	18.52	450.54
Hydraulic Shovel (34 m^3)	44.00	-	-	-	187.15	22.22	253.37
Haulage Truck – 229 t	187.96	18.80	47.79	-	94.30	4.44	353.29
Haulage Truck – 91t	37.59	3.76	15.58	-	46.53	2.22	105.68
Haulage Truck – 40t	32.22	3.22	11.80	-	33.53	-	80.78
Crusher Loader	80.56	8.06	21.34	-	64.70	7.41	182.06
Track Dozer	80.56	8.06	-	7.41	37.87	3.70	137.59
Grader	21.48	2.15	9.14	-	3537	3.70	71.84

Drilling in the open pit will be performed using conventional down the hole (DTH) blasthole rigs with 165- and 203-mm bits. The pattern size was the same for mill feed and waste and are blasted with recognition that the rock is competent, and finer material improves productivity and reduces maintenance costs as well as improved plant performance. The drill pattern parameters are shown in Table 21-18.

Table 21-18: Drill Pattern Specification

Specification	Unit	Mill Feed/Waste (165 mm)	Mill Feed/Waste (203 mm)
Bench Height	m	10	10
Sub-Drill	m	0.9	1.1
Blasthole Diameter	mm	165	203
Pattern Spacing – Staggered	m	5.1	6.1
Pattern Burden – Staggered	m	4.6	5.5
Hole Depth	m	10.9	11.1

The sub-drill was included to allow for caving of the holes in the weaker zones, avoiding re-drilling of the holes or short holes that would affect bench floor conditions and thereby increasing tire and overall maintenance costs.

Below in Table 21-19 are the parameters used for estimating drill productivity. The drill is configured for single pass drilling of the 11.1 m deep hole.

Table 21-19: Drill Productivity Criteria

Drill Activity	Unit	Mill Feed/Waste (165 mm)	Mill Feed/Waste (203 mm)
Pure Penetration Rate	m/min	0.55	0.50
Hole Depth	m	10.9	11.1
Drill Time	min	19.82	22.20
Move, Spot, and Collar Blasthole	min	3.00	3.00
Level Drill	min	0.50	0.50
Add Steel	min	0.50	0.00
Pull Drill Rods	min	1.50	1.00
Total Setup/Breakdown Time	min	5.50	4.50
Total Drill Time per Hole	min	25.3	26.7
Drill Productivity	m/h	25.8	24.9

An emulsion product will be used for blasting to provide water protection. With the wet conditions expected, it is believed that a water-resistant explosive will be required. The powder factors used in the explosives calculation are shown in Table 21-20.

Table 21-20: Design Powder Factors

	Unit	Mill Feed/Waste (165 mm)	Mill Feed/Waste (203 mm)
Powder Factor	kg/m^3	0.84	0.84
Powder Factor	kg/t	0.31	0.31

The blasting cost is estimated using quotations from a local vendor. The emulsion price is $85.20/100 kg. The mine is responsible for guiding the loading process, including placement of boosters/Nonels, and stemming and firing the shot.

Total monthly cost in the service of delivering the explosives to the hole is $284,000/month for the vendor’s pickup trucks, pumps, and labour is also applied and covers the cost of the explosives plant. The explosives vendor also leases the explosives and accessories magazines to Troilus as part of that cost.

Mill feed and waste loading costs were estimated using the front-end loaders and hydraulic shovels as the only loading units. The shovels are the primary diggers for mill feed and waste, with the front-end loaders being used as backup. The average percentage of each material type that the various loading units are responsible for is shown in Table 21-21. This highlights the focus on the shovels over the loaders.

Table 21-21: Loading Parameters – Year 5

	Unit	Front-End Loader	Hydraulic Shovel
Bucket Capacity	m^3	23	37
Waste Tonnage Loaded	%	10	90
Mill Feed Tonnage Mined	%	10	90
Bucket Fill Factor	%	91	93
Cycle Time	sec	42	38
Trucks Present at the Loading Unit	%	80	80
Loading Time	min	4.2	2.6

The trucks present at the loading unit refers to the percentage of time a truck is available to be loaded. To maximize truck productivity and reduce operating costs, it is more efficient to slightly under-truck the loader or shovel. The single largest operating cost item is haulage and minimizing this cost by maximizing truck productivity is crucial to lower operating costs. The value of 80% comes from the standby time shovels typically encounter due to a lack of trucks.

Haulage profiles were determined for each pit phase for the primary crusher or the waste rock management facility destinations. Cycle times were generated for the appropriate period tonnage by destination and phase to estimate the haulage costs. Maximum speed on trucks is limited to 50 km/h for tire life and safety reasons. Calculation speeds for various segments are shown in Table 21-22.

Table 21-22: Haulage Cycle Speeds

	Flat (0%) on surface	Flat (0%) Inpit, Crusher, Dump	Slope Up (8%)	Slope Up (10%)	Slope Down (8%)	Slope Down (10%)	Acceleration or Deceleration
Loaded (km/h)	50	40	14	12	28	23	20
Empty (km/h)	50	40	35	25	35	35	20

Support equipment hours and costs are determined using the percentages shown in Table 21-23.

Table 21-23: Support Equipment Operating Factors

Mine Equipment	Factor	Factor Units
Track Dozer	25%	Of haulage hours to a maximum of 6 dozers
Grader	15%	Of haulage hours to a maximum of 3 graders
Crusher Loader	40%	Of loading hours to maximum of 1 loader
Support Backhoe	30%	Of loading hours to maximum of 2 backhoes
Water Truck	10%	Of haulage hours to a maximum of 2 trucks
Lube/Fuel Truck	8	h/d
Mechanic's Truck	14	h/d
Welding Truck	8	h/d
Blasting Loader	8	h/d
Blaster's Truck	8	h/d
Integrated Tool Carrier	4	h/d
Compactor	4	h/d
Lighting Plants	12	h/d
Pickup Trucks	10	h/d
Dump Truck – 20 ton	6	h/d

These percentages resulted in the need for six track dozers, three graders, and two support backhoes. Part of this is due to the spread-out nature of the various pit areas which landlocks some of the equipment for periods of time. Their tasks include cleanup of the loader faces, roads, dumps, and blast patterns. The graders will maintain the mill feed and waste haul routes. In addition, water trucks have the responsibility for patrolling the haul roads and controlling fugitive dust for safety and environmental reasons. The support backhoes will assist on dilution control on mill feed/waste separation. A small backhoe will be responsible for cleaning out sedimentation ponds and water ditch repairs together with the two small dump trucks.

These hours are applied to the individual operating costs for each piece of equipment. Many of these units are support equipment so no direct labour force is allocated to them due to their function.

Grade Control

Grade control will be completed with a separate fleet of reverse circulation (RC) drill rigs. They will drill the deposit off on a 10 m x 5 m pattern in areas of known mineralization taking samples each metre. The holes will be inclined at 60°.

In areas of low-grade mineralization or waste the pattern spacing will be 20 m x 10 m with sampling over 5 m. These holes will be used to find undiscovered veinlets or pockets of mineralization. Over the life of the mine, a total of 3.0 Mm of drilling are expected to be completed for grade control work.

A total of 3.3 million samples will be assayed from that drilling at a cost of $14.25/sample. Samples collected will be sent to the assay laboratory and assayed for use in the short-range mining model.

Costs associated with this separate drill program are tracked as a distinct line item for the mining cost. Each drill crew is one driller and two helpers with oversight by the Mine Geology department. Two drills will be required for the project. The cost of this drilling is expected to be on average over $6 million per year with a peak of $8.7 million in Year 10.

Finance Cost

Financing of the mine fleet was investigated with the major vendors and is considered a viable option to reduce initial capital. Various vendors offer this as an option to help select their equipment. Caterpillar, Komatsu and Liebherr have the ability, and desire, to allow financing of their product lines.

Indicative terms for leasing provided by the vendors are:

- Down Payment = 10% of equipment cost – if the entire fleet selection was one vendor
- Term Length = variable between 2 and 5 years depending on equipment type
- Interest Rate = CORRA plus a percentage
- Residual = $0

The proposed interest rate is used to calculate a multiplier on the amount being leased. The multiplier is 1.26 to equate to the rate. It does not consider a declining balance on the interest but rather the full amount of interest paid over finance period, equally distributed over the equipment finance years.

As a 10% downpayment was required, the downpayment was allocated to capital. The remainder of the financing cost is being borne as an operating cost.

All of the major mine equipment, and the large majority of the support equipment where it was considered reasonable, was financed. If the equipment has a life greater than the finance period, then the following years onwards of the equipment does not have a finance payment applied. In the case of the mine trucks, with an approximate 10 year working life, the finance period would be complete in 5 years and the trucks would simply incur operating costs after that time. For this reason, the operating cost would vary annually depending on the equipment replacement schedule and timing of the financing.

Utilizing the financing option adds $ 0.51/t to the mine operating cost over the life of the mine. On a cost per tonne of mill feed basis, it was $ 1.92/t mill feed.

Dewatering

Pit dewatering is an important part of mining at Troilus particularly since the pits will be below the creek level and is currently full of water. Efficient and cost-effective dewatering will play a role in the Project development. Dewatered slopes may allow a reduction in the strip ratio by permitting steeper inter-ramp angles that would also be inherently safer.

It is estimated that 7.4 Mm^3/year on average will need to be pumped from within the pits. From there, it will need to be pumped to the required discharge point near the settling ponds. Storm events have the potential to impact mining operations, and an estimate of 44,800 m^3/d of pumping may be required for a short period of time to recover from one of these storm events. The capital cost estimate has considered this in the calculation for the number of pumps required on site to handle such an event.

The dewatering system includes the pumps, sumps, and pipelines responsible for moving water from the pit to the discharge points. Labour for this is already included in the General and Mine Engineering category of the mine operating cost. The mine has a dedicated pump crew and pump crew foreman.

Additional dewatering in the form of horizontal drain holes is also part of the dewatering operating costs. These holes will be drilled in annual campaigns starting in Year 2. The design concept is a series of holes 50 m in length, angled up slightly and drilled into the highwalls. They will allow the water behind the wall to drain freely and prevent pore water pressure buildup particularly during freezing conditions.

The horizontal drill holes are considered as a capital cost for a total of $11.2 million over the life of the mine.

The dewatering operating cost is estimated at $8.0 million over the mine life or $0.4 million/a.

Total Open Pit Mine Costs

The total life of mine operating costs per tonne of material moved and per tonne of mill feed processed are shown in Table 22-24 and Table 22-25.

Table 21-24: Open Pit Mine Operating Costs – with Finance Cost ($/t Total Material)

Open Pit Operating Category	Unit	Year 1	Year 3	Year 5	LOM Average Cost
General Mine and Engineering	$/t	0.16	0.16	0.15	0.18
Drilling	$/t	0.18	0.23	0.22	0.23
Blasting	$/t	0.34	0.39	0.37	0.41
Loading	$/t	0.17	0.13	0.13	0.15
Hauling	$/t	0.84	0.96	1.14	1.29
Support	$/t	0.20	0.20	0.19	0.23
Grade Control	$/t	0.04	0.05	0.07	0.07
Finance Costs	$/t	0.71	0.78	0.38	0.51
Dewatering	$/t	0.00	0.01	0.02	0.01
Total	**$/t**	**2.65**	**2.91**	**2.66**	**2.96**

Table 21-25: Open Pit Mine Operating Costs – with Finance Cost ($/t Mill Feed)

Open Pit Operating Category	Unit	Year 1	Year 3	Year 5	LOM Average Cost
General Mine and Engineering	$/t mill feed	1.00	0.70	0.71	0.68
Drilling	$/t mill feed	1.13	1.00	1.02	0.88
Blasting	$/t mill feed	2.10	1.70	1.72	1.55
Loading	$/t mill feed	1.05	0.57	0.60	0.56
Hauling	$/t mill feed	5.23	4.21	5.38	4.84
Support	$/t mill feed	1.27	0.88	0.88	0.87
Grade Control	$/t mill feed	0.26	0.22	0.33	0.25
Finance Costs	$/t mill feed	4.42	3.44	1.79	1.92
Dewatering	$/t mill feed	0.02	0.05	0.10	0.05
Total	**$/t mill feed**	**16.47**	**12.76**	**12.52**	**11.60**

21.2.3 Process Operating Costs

The process plant operating costs have been developed based on a design processing rate of 50 kt/d or 18.25 Mt/a of ore. The process plant will normally operate 24 h/d, 365 d/a with 6,570 h/a for the crushing circuit and 7,709 h/a for grinding and flotation circuits. All costs to an accuracy of +/-15% and are based on Q2 2024 pricing. The process plant operating costs are broken down into fixed and variable costs and are summarized below in Table 21-26. No contingency has been added to the process operating cost.

Table 21-26: Process Operating Costs

Cost Centre	Total M$/a	Fixed M$/a	Variable M$/a	Total $/t	Fixed $/t	Variable $/t	Proportion of Operating Total %	Proportion of Operating Fixed %	Proportion of Operating Variable %
Scenario Without Gravity Circuit									
Operating Consumables	63.7	0	63.7	3.49	0.00	3.49	62.9	0.0	62.9
Plant Maintenance	10.7	4.6	6.1	0.59	0.25	0.33	10.5	4.6	6.0
Laboratory	0.1	0.1	0	0.00	0.00	0.00	0.1	0.1	0.0
Power	15.9	15.9	0	0.87	0.87	0.00	15.7	15.7	0.0
Labour	11.0	11.0	0	0.60	0.60	0.00	10.8	10.8	0.0
Total Process Plant	**101.4**	**31.6**	**69.8**	**5.56**	**1.73**	**3.82**	**100**	**31.2**	**68.8**
Scenario Without Gravity Circuit									
Operating Consumables	64.2	0	64.2	3.52	0.00	3.52	62.4%	0.0%	62.4%
Plant Maintenance	10.9	4.7	6.2	0.60	0.26	0.34	10.5%	4.6%	6.0%
Laboratory	0.3	0.3	0	0.02	0.02	0.00	0.3%	0.3%	0.0%
Power	16,1	15.9	0.2	0.88	0.87	0.01	15.6%	15.4%	0.2%
Labour	11.4	11.4	0	0.63	0.63	0.00	11.1%	11.1%	0.0%
Total Process Plant	**103.0**	**32.4**	**70.6**	**5.64**	**1.77**	**3.87**	**100.0%**	**31.4%**	**68.6%**

The process operating costs were developed in accordance with industry practice for a copper and gold processing plant. Quantities and cost data were compiled from a variety of sources including:

- metallurgical testwork
- consumable prices from suppliers
- Lycopodium internal data
- first principal calculations

The following major categories were used to estimate the process operating cost:

- operating consumables (reagents, steel, fuel, tools, and safety supplies)
- plant maintenance costs
- power
- labour (operations and maintenance)
- laboratory costs

Operating Consumables

The consumables category includes reagents, fuel, and operating consumables such as crusher /mill liners, grinding media, cyclone parts, screen panels, HPGR tyres replacement, mill lubricants, and concentrate filter consumables. It excludes general maintenance consumables such as greases and lubricants, equipment spare parts, and pump wear parts which are covered in maintenance costs. Consumption rates and pricing for consumables and reagents have been estimated based on the following:

- Comminution consumables include crusher, HPGR and mill liners; grinding media and were predicted based on the material bond abrasion index and the mill power consumption.
- Reagent consumptions were derived from metallurgical testwork.
- Fuel consumption for mobile equipment is based on standard fuel consumption rates and equipment utilization.
- Reagent prices were derived from reagent supplier quotes.

Maintenance

Maintenance costs exclusive of labour and consumable costs were estimated as a factor of capital costs. A factor of 1 to 2% was used and is consistent with Lycopodium experience on similar projects.

Power

Process plant power consumption is based on the installed motor size of individual unit items of equipment, excluding standby equipment adjusted by efficiency, load, and utilisation factors to derive an annual average power draw. This is then multiplied by the total hours of operation per annum and the electricity price to obtain the total power cost.

The overall average power consumption is estimated at 612.9 MWh/a for the scenario without gravity circuit and 620.4 MWh/a for the future scenario with gravity circuit. The power cost for the scenario without and with gravity circuit was estimated at $15.9 million dollars/a and $16.1 million dollars/a, respectively.

Labour

The process plant operating, and maintenance labour costs were estimated from first principles based on a typical organization chart and typical labour rates from Normandin Beaudry direct compensation report for 2023 for the mining industry sector. The process plant labour includes a combination of day and shift work. A summary of the labour complement is provided below in Table 21-27.

Table 21-27: Process Labour

Location	Without Gravity Circuit - Number of Employees	With Gravity Circuit - Number of Employees
Operations	62	67
Maintenance	49	49
Laboratory	13	13
Total	**124**	**129**

The following shift rotations were assumed:

- professional employees and management - 4 days on/ 3 days off
- operations and maintenance staff - 12-hour shifts, 4 days on, 4 days off rotation

Laboratory Costs

Laboratory costs typically include the necessary plant samples to monitor metallurgical performance and include sample preparation, fire assay, size analysis and chemical analyses of production samples. Grade control costs are captured under mining. The average cost is under $0.01/t ore processed for the scenario with no gravity circuit, and $0.02/t ore processed for future scenario with gravity circuit included.

Exclusions

The process operating cost estimate excludes the following items:

- escalation past Q2 2024
- ROM and material handling costs
- all G&A costs
- bullion refining and transport costs
- bullion in transit insurance
- environmental monitoring and compliance costs
- closure costs
- local and federal government taxes and duties
- impacts of foreign exchange rates
- licence and union fees
- other insurance costs
- contingency

21.2.4 General and Administrative Operating Costs

The General and Administrative costs were estimated for each year of the Project schedule. This was to consider the varying manpower levels for the camp cost and overall G&A needs. The G&A costs peaked in Year 5 at $22.6 million/a.

A catering and accommodation quotation of $51 per day per person in camp was provided by a local vendor. This cost was the single largest cost in the G&A amounting to $8.0 million per year in Year 6. From that point in time the camp cost declined as the open pit feed rates dropped and personnel in the mine started to drop from the peak.

The camp vendor provided the option to lease the permanent camp, reducing the need to purchase it upfront. The cost of the permanent camp delivered to the site is estimated at $12.5 million and with the leasing cost totaled $12.5 million for the life of mine. This was distributed as a cost in the G&A for the first 5 years at $2.5 million/a to pay off the camp.

Wages for staff and hourly personnel in the G&A area totaled $ 6.2 million/a.

Bussing from Chibougamau and other areas of personnel peaked at $ 1.1 million/a in Year 6.

The life of mine average G&A cost was $ 1.14/t of ore or $ 430.5 million total life of mine.

21.2.5 Concentrate Trucking, Port Costs, and Shipping to Smelter

The cost of trucking was determined by contacting various rail and trucking companies and also discussions with the metal trading firms providing smelter terms. From these inquiries, an estimate of $72.50/WMT of concentrate was used in the calculations with the assumption of trucking to Montreal for shipment overseas to smelters.

Port storage and handling costs are estimated at $20 per wet metric tonne.

Shipment costs are assumed with a destination to Germany and a cost of $65/WMT provided by the metal trading firm as part of the overall term sheet.

The life of mine operating cost estimate summary is shown in Table 21-28.

Table 21-28: Troilus Project Operating Cost Estimate

Area	Units	Year 1 – 5	Year 6 – 22	Life of Mine (Year 1-22)
Open Pit Mining	$/t moved	2.82	3.01	2.96
	$/t mill feed	13.33	11.09	11.60
Processing	$/t mill feed	5.63	5.64	5.64
G&A	$/t mill feed	1.28	1.09	1.14
Concentrate Trucking, Port, Shipping	$/t mill feed	0.68	0.68	0.68
Total Operating Cost	**$/t mill feed**	**20.92**	**18.51**	**19.06**

22 ECONOMIC ANALYSIS

22.1 Introduction

A pre-tax and post-tax cash flow model has been prepared by AGP on behalf of Troilus Gold for the evaluation of the Project Feasibility Study (FS). The model was created in Excel for Troilus' use.

The FS was prepared in accordance with NI 43-101.

The mineral resource currently has no Proven mineral reserves material therefore only Probable reserves reported in Section 15 were used in the economic analysis of the Project. The quantity used in the economic analysis is shown in Table 22-1

Table 22-1: Troilus Mine Reserves

Mining Type	Area	Probable					
		Reserves (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	AuEq (g/t)	CuEq (%)
Open Pit	Z87	166	0.55	0.062	1.12	0.66	0.43
	J	125	0.44	0.058	0.88	0.54	0.36
	X22	36	0.41	0.058	1.16	0.52	0.34
	SW	52	0.49	0.045	0.76	0.58	0.35
Total		**380**	**0.49**	**0.058**	**1.00**	**0.59**	**0.39**

All pricing is H2 2023 United States dollars unless otherwise noted.

The mine is envisaged as an open pit only operation utilizing four separate pit areas with multiple phases in each pit.

The results of the financial analysis using discounted cash flow is summarized in Table 22-2. All calculations and data were collected in United States dollars.

Table 22-2: Troilus Gold Project – Discounted Cash Flow Financial Summary (US$)

Parameter	Units	Pre-Tax	Post-Tax
Metal Prices			
Gold	US$/oz	1,975	
Copper	US$/lb	4.05	
Silver	US$/oz	23	
Exchange Rate	C$:US$	0.74	
Net Present Value (5%)	US$ M	1,564	884.5
Internal Rate of Return	%	18.1	14.0
Net Revenue less Royalties	US$ M	12,122.0	12,122.0
Total Operating Cost	US$ M	7,224.2[1]	7,224.2[1]
Life of Mine Capital Cost	US$ M	1,351.2	1,351.2
Taxes	US$ M	-	1,342.0
Net Cash Flow	US$ M	3,546.5	2,204.6
Payback Period	Years	5.4	5.7
Cash Costs (with credits)	US$/oz	1,064	1,313
All-in Sustaining Cost	US$/oz	1,109	
Payable Metals (Life of Mine)			
Gold	Moz	5.38	
Copper	M Lbs	381.8	
Silver	Moz	9.45	
Initial Capital	US$ M	1,074.6	
Sustaining Capital	US$ M	276.6	
Total Capital	US$ M	1,351.2	
Mine Life	Years	21.1	

[1] Includes the processing cost in Year -1 which is not capitalized.

22.2 Discounted Cash Flow Analysis

The Discounted Cash Flow (DCF) analysis was completed using the Base Case Parameters shown in Table 22-3.

Table 22-3: Discounted Cash Flow – Parameters

Parameter	Units	Value
Metal Prices		
Gold	US$/oz	1,975
Copper	US$/lb	4.05
Silver	US$/oz	23
Exchange Rate	C$:US$	0.74
Royalties – all metals	%	1.0
Net Present Value Discount Rate	%	5.0

22.3 Mineral Reserve and Mine Life

The mineral reserves used in the analysis are from the four open pits discussed in Section 15. The tonnage and grades are shown again in Table 22-4.

Table 22-4: Mine Feed Tonnages and Grade

Item	Unit	Open Pit
Mill Feed	Mt	379.5
Gold grade	g/t	0.49
Copper Grade	%	0.06
Silver Grade	g/t	1.00
Waste	Mt	1,170.9
Life of Mine Strip Ratio	Waste:Mill Feed	3.1

22.4 Metallurgical Recoveries and Concentrate Grades

Metallurgical recoveries used in the analysis are dependent on the area mined and the grade of the incoming feed. Copper is recovered in a copper concentrate with gold and silver. Gold and silver also report to a dore from the gravity gold circuit after Year 1. The concentrate grades by area are determined by fixed tails assays and the mass pull to the concentrate. The weighted average of the feed origin is used to determine the recoveries and the appropriate concentrate grade for each year.

The information for the project is summarized in Table 22-5. The soluble loss was estimated at 0.03%.

Table 22-5: Metallurgical Recoveries and Concentrate Mass Pulls by Zone

Item	Unit	87 Zone	J Zone	SW Zone	X22 Zone
Tails Assay					
Gold	g/t	0.029	0.036	0.062	0.028
Copper	%	0.005	0.005	0.005	0.003
Silver	g/t	0.027	0.119	0.131	0.117
Mass Pull to Concentrate	% of Feed	0.400	0.400	0.400	0.400
Dore Recovery					
Gold	%	31.7	34.9	21.1	34.9
Silver	%	3.8	4.1	2.6	4.1

22.4.1 Smelting and Refining Terms

The smelting and refining terms used in the analysis reflect current market conditions and the expected range of copper concentrate grades from Troilus. The Life of Mine average is shown in Table 22-6 and the variable terms are shown in Table 22-7.

Table 22-6: Smelting and Refining Terms - LOM

Term	Unit	Copper	Gold	Silver
Cu, Au Minimum Deduction	%, g/dmt	1.5%	2.0 g/dmt	20.0 g/dmt
Base Smelting Charge	US$/dmt	77.00	-	
Cu Refining Charge	US$/lb payable	0.077	-	
Payable	%	96.6	98.0	97.0
Refining Charge (Gold, silver)	US$/oz	-	5.00	0.50
Concentrate Grade (LOM average)	%, g/t	13.3%	80 gpt	229 gpt
Concentrate Moisture	%	8.0	-	

Table 22-7: Copper Concentrate Variable Terms

Concentrate Grade (Cu%)	Copper Payable (%)	Copper Deduction (%Cu units)
3% - 12%	96.6	1.5 units
12% - 14%	96.6	1.5 units
14% - 16%	96.6	1.4 units
16% - 18%	96.6	1.3 units
18% - 20%	96.6	1.2 units
20%+	96.6	1.1 units

For the gold dore produced from the gravity concentrator, the terms applied are shown in Table 22-8.

Table 22-8: Dore Terms

Term	Unit	Gold	Silver
Dore Payable	%	99.98	100.00
Dore Selling Cost	US$/oz	1.00	0.65

22.4.2 Operating Costs

The mining, processing, administration and concentrate transportation costs are based on the operating cost estimates presented in Section 21. A summary of the costs is shown in Table 22-9.

Table 22-9: Troilus Cash Flow – Life of Mine Operating Cost Summary

Cost Area	Cost (M$)	Unit Cost ($/t Mill Feed)
Open Pit Mining	4,394.5	11.6
Processing	2,135.4	5.64
General and Administration	430.5	1.14
Concentrate Trucking	119.4	0.32
Port Costs	32.9	0.09
Shipping to Smelter	107.1	0.28
Total Operating Cost	**7,079.8**	**19.06**

22.4.3 Capital Costs

The capital costs by area used in the discounted cash flow are the same as presented in Section 21. They are summarized again by area in Table 22-10.

Table 22-10: Troilus Cash Flow – Mine Capital Cost Summary

Area	Initial Capital (M$)	Sustaining Capital (M$)	Total Capital (M$)
Open Pit – Prestrip (capitalized)	213.0	-	213.0
Open Pit - Capital	45.3	99.3	144.6
Open Pit Mining - Subtotal	258.3	99.3	357.6
Processing	443.0	15.1	458.1
Infrastructure	100.3	27.7	128.0
Environmental	10.7	67.4	78.2
Indirects	173.0	50.5	223.4
Contingency	89.3	16.6	105.9
Total	**1,074.6**	**276.6**	**1,351.2**

22.4.4 Royalties

Royalties on all the metals is set at 1.0% per current Troilus Gold agreements.

22.4.5 Depreciation

All capital expenditures are depreciated depending on the class of asset.

22.4.6 Taxes

A taxation model was developed that considered the proper taxation for the jurisdiction of Québec as well as Federal Tax. Estimates for the Canadian Development Expenses (CDE), the Canadian Exploration Expenses (CEE) and Capital Cost Allowance (CCA) were incorporated into the cash flow model to properly model taxation on an annual basis for the project schedule.

The LOM taxes determined were:

1) Québec Taxes = $ 909.7 million
2) Federal Taxes = $ 432.3 million
3) Total Taxes = $ 1,342.0 million

The amount of tax is approximately 37.8% of the pre-tax cash flow. Taxation added 0.3 years to the payback period of the project.

22.4.7 General

The financial analysis of the project has several assumptions:

- assumes full equity funding
- no provision for corporate head office costs during operations

- inflation or escalation is not considered

22.5 Sensitivity Analysis

The project value was assessed by undertaking a sensitivity analysis on metal prices, operating and capital costs and exchange rate. The results of the sensitivity analyses are presented in Table 22-11 and Table 22-12.

Table 22-11: NPV Sensitivity (Post-Tax) ($M)

NPV $ M	-20%	-10%	Base Case	+10%	+20%
Metal Price	17.3	460.9	885	1,288.8	1,686.1
Capital Cost	1,051.1	967.8	885	801.2	717.9
Operating Cost	1,335.7	1,113.9	885	643.2	391.1

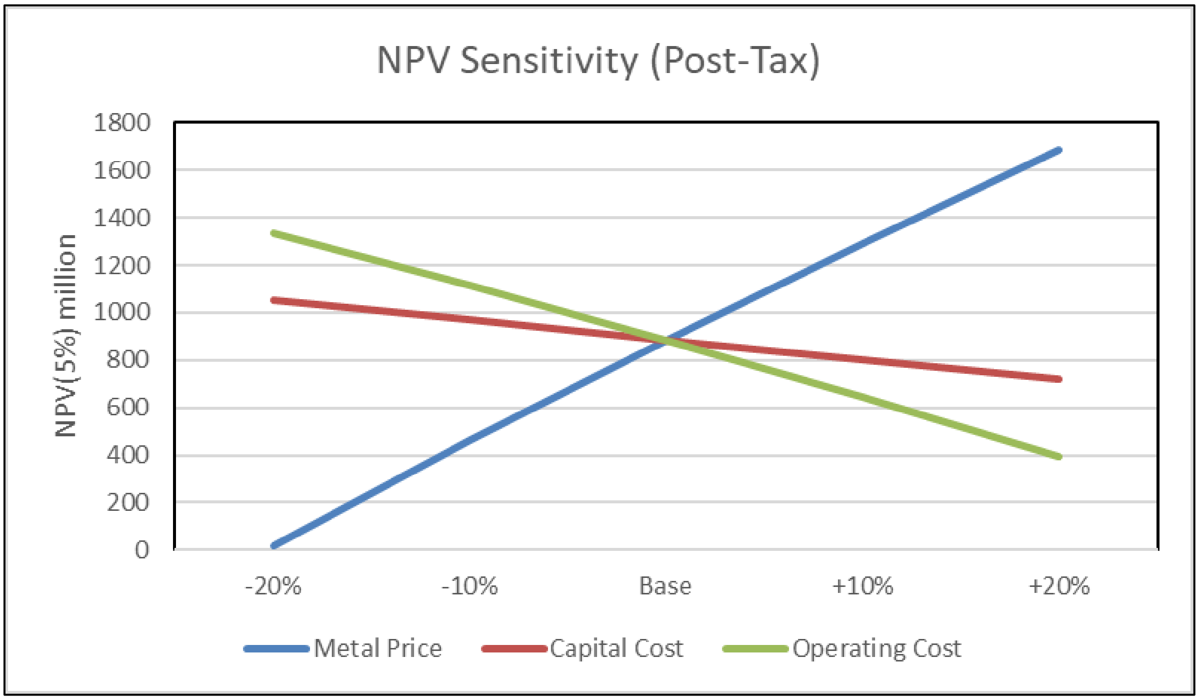
Table 22-12: IRR Sensitivity (Post-Tax)

IRR %	-20%	-10%	Base Case	+10%	+20%
Metal Price	5.2	10.1	14.0	17.4	20.5
Capital Cost	16.8	15.3	14.0	12.8	11.7
Operating Cost	17.9	16.0	14.0	11.8	9.4

The project is most sensitive to changes in metal prices. Sensitivity to metal pricing is the most sensitive aspect of the Project. Within the 20% variation examined, the change in metal prices can increase the project NPV value by $800 million or reduce it to $17 million if the base case metal price were to drop by 20%. Capital and operating costs also are impacted within the +/- 20% window but to a maximum NPV variation of -$165 million or +$450 million significantly less than the impact metal prices have.

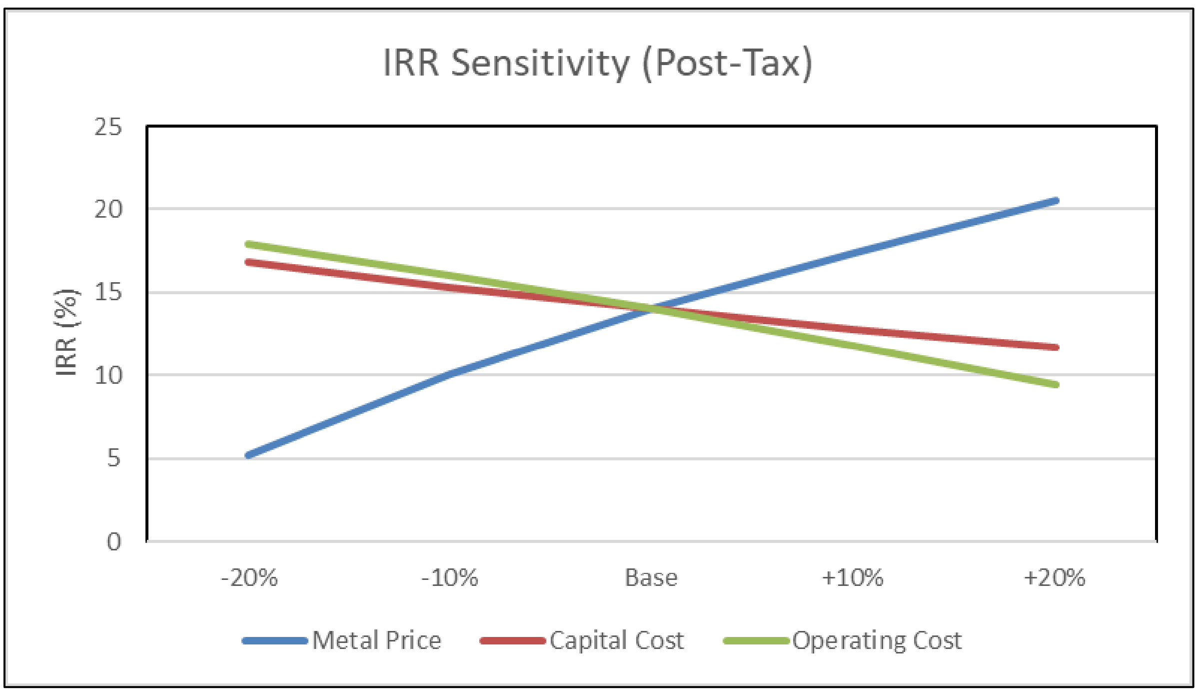
The sensitivities are also show graphically in Figure 22-1 and Figure 22-2.

Figure 22-1: NPV Sensitivity (Post-Tax)



Source: AGP (2024)

Figure 22-2: IRR Sensitivity (Post-Tax)



Source: AGP (2024)

22.6 Detailed Cash Flow Sheets

The Base Case cash flow has been included in Table 22-13 to Table 22-16. This includes the mine schedule, concentrate calculations, operating and capital costs, revenue estimates and cumulative cash flow on which the NPV and IRR have been determined. This is on both a pre-tax and post-tax basis.

Table 22-13: Detailed Cash Flow – Mill Production Calculations (Year -3 to Year 10)

		Total			Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Mill Production																	
Total Mill Feed	tonnes	379,518,417				-	714,286	12,857,142	18,249,996	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000
Gold	gpt	0.493				-	0.36	0.52	0.38	0.43	0.54	0.72	0.63	0.84	0.63	0.49	0.56
Copper	%	0.058				-	0.06	0.04	0.04	0.05	0.06	0.08	0.06	0.09	0.08	0.06	0.06
Silver	gpt	0.996				-	1.23	1.02	0.85	0.93	1.11	1.23	1.07	1.27	1.05	0.90	1.08
87 Pit Feed	**tonnes**	**166,084,772**					**714,286**	**9,666,384**	**12,352,085**	**11,477,859**	**12,798,145**	**15,153,919**	**10,184,645**	**11,735,318**	**5,064,517**	**-**	**-**
Percentage of feed tonnes	%	43.8%					100.0%	75.2%	67.7%	62.9%	70.1%	83.0%	55.8%	64.3%	27.8%	0.0%	0.0%
Gold	gpt	0.55					0.36	0.53	0.36	0.41	0.55	0.75	0.69	0.95	0.99	-	-
Copper	%	0.06					0.06	0.05	0.04	0.05	0.06	0.08	0.08	0.11	0.11	-	-
Silver	gpt	1.119					1.23	1.14	0.95	1.08	1.26	1.29	1.24	1.42	1.41	-	-
J Pit	**tonnes**	**125,164,622**					**-**	**-**	**-**	**-**	**-**	**326,334**	**1,588,527**	**2,843,598**	**7,239,851**	**14,477,244**	**18,250,000**
Percentage of feed tonnes	%	33.0%					0.0%	0.0%	0.0%	0.0%	0.0%	1.8%	8.7%	15.6%	39.7%	79.3%	100.0%
Gold	gpt	0.44					-	-	-	-	-	0.63	0.48	0.56	0.41	0.45	0.56
Copper	%	0.06					-	-	-	-	-	0.07	0.07	0.07	0.07	0.06	0.06
Silver	gpt	0.883					-	-	-	-	-	1.26	1.09	1.03	0.91	0.91	1.08
SW Pit	**tonnes**	**51,864,267**					**-**	**3,190,758**	**5,897,911**	**6,772,141**	**5,451,855**	**2,769,747**	**6,476,827**	**3,671,084**	**5,945,632**	**3,772,756**	**-**
Percentage of feed tonnes	%	13.7%					0.0%	24.8%	32.3%	37.1%	29.9%	15.2%	35.5%	20.1%	32.6%	20.7%	0.0%
Gold	gpt	0.49					-	0.48	0.42	0.45	0.53	0.60	0.55	0.70	0.58	0.65	-
Copper	%	0.05					-	0.04	0.04	0.04	0.05	0.05	0.03	0.05	0.06	0.07	-
Silver	gpt	0.758					-	0.65	0.65	0.67	0.75	0.90	0.80	0.97	0.90	0.88	-
X22 Pit	**tonnes**	**36,404,756**					**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Percentage of feed tonnes	%	9.6%					0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Gold	gpt	0.41					-	-	-	-	-	-	-	-	-	-	-
Copper	%	0.06					-	-	-	-	-	-	-	-	-	-	-
Silver	gpt	1.16					-	-	-	-	-	-	-	-	-	-	-
Combined Tail Assay - Weighted																	
Gold							0.029	0.037	0.040	0.041	0.039	0.034	0.041	0.037	0.042	0.041	0.036
Copper							0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005
Silver							0.027	0.053	0.061	0.066	0.058	0.045	0.072	0.062	0.097	0.121	0.119
Combined Recovery																	
Gold							91.9%	92.8%	89.6%	90.3%	92.8%	95.3%	93.4%	95.6%	93.2%	91.7%	93.6%
Copper							91.5%	89.1%	88.3%	89.8%	91.6%	93.7%	92.5%	94.6%	93.7%	92.4%	92.0%
Silver							97.8%	94.8%	92.9%	92.9%	94.7%	96.4%	93.3%	95.1%	90.7%	86.6%	89.0%
Recovered Metals																	
Gold		New															
Total Gold Recovered	ounces	5,580,993					7,644	199,072	201,386	226,889	295,223	404,880	343,275	468,850	342,261	265,546	307,642
Gravity Percentage	%	28.3%					0.0%	0.0%	28.3%	27.8%	28.5%	30.1%	28.2%	30.1%	29.5%	32.0%	34.9%
Gold - Dore	ounces	1,694,574					-	-	56,924	62,980	84,212	122,029	96,832	140,924	100,992	85,075	107,335
Gold - Concentrate	ounces	3,886,418					7,644	199,072	144,463	163,909	211,011	282,852	246,444	327,926	241,269	180,472	200,307
Copper																	
Copper Recovered	tonnes	201,622					386	5,155	6,693	7,788	9,725	13,497	10,877	15,786	13,173	10,704	10,225
	pounds	444,495,765					851,559	11,364,162	14,755,542	17,169,763	21,440,096	29,755,662	23,979,087	34,802,759	29,041,280	23,598,576	22,541,763
Silver																	
Total Silver Recovered	ounces	11,168,597					27,538	399,688	464,584	505,979	615,129	694,227	586,130	708,265	557,358	457,916	564,171
Gravity Percentage	%	3.4%					0.0%	0.0%	3.4%	3.4%	3.4%	3.6%	3.4%	3.6%	3.5%	3.8%	4.1%
Silver - Dore	ounces	404,355					-	-	15,848	16,969	21,163	25,146	19,924	25,528	19,658	17,349	23,124
Silver - Concentrate	ounces	10,764,242					27,538	399,688	448,736	489,010	593,966	669,081	566,206	682,738	537,700	440,566	541,046
Total																	
Gold - Dore	ounces	1,694,574			-	-	-	-	56,924	62,980	84,212	122,029	96,832	140,924	100,992	85,075	107,335
Gold - Flotation	ounces	3,886,418			-	-	7,644	199,072	144,463	163,909	211,011	282,852	246,444	327,926	241,269	180,472	200,307
Gold Total	*ounces*	*5,580,993*			-	-	*7,644*	*199,072*	*201,386*	*226,889*	*295,223*	*404,880*	*343,275*	*468,850*	*342,261*	*265,546*	*307,642*
Copper	pounds	444,495,765			-	-	851,559	11,364,162	14,755,542	17,169,763	21,440,096	29,755,662	23,979,087	34,802,759	29,041,280	23,598,576	22,541,763
Silver - Dore	ounces	404,355			-	-	-	-	15,848	16,969	21,163	25,146	19,924	25,528	19,658	17,349	23,124
Silver - Concentrate	ounces	10,764,242			-	-	27,538	399,688	448,736	489,010	593,966	669,081	566,206	682,738	537,700	440,566	541,046
Silver Total	*ounces*	*11,168,597*			-	-	*27,538*	*399,688*	*464,584*	*505,979*	*615,129*	*694,227*	*586,130*	*708,265*	*557,358*	*457,916*	*564,171*
Concentrate Yield		%					0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40
Cu Concentrate (DMT) - Grades	DMT	1,518,074					2,857	51,429	73,000	73,000	73,000	73,000	73,000	73,000	73,000	73,000	73,000
Gold	g/DMT	79.6					83.2	120.4	61.6	69.8	89.9	120.5	105.0	139.7	102.8	76.9	85.3
Copper	%	13.3%					13.5%	10.0%	9.2%	10.7%	13.3%	18.5%	14.9%	21.6%	18.0%	14.7%	14.0%
Silver	g/DMT	228.8					299.8	241.7	191.2	208.4	253.1	285.1	241.2	290.9	229.1	187.7	230.5
Cu Concentrate (WMT)	WMT	1,650,080					3,106	55,901	79,348	79,348	79,348	79,348	79,348	79,348	79,348	79,348	79,348
Cu Concentrate Delivered to Smelter (less losses)	DMT	1,510,483					2,843	51,171	72,635	72,635	72,635	72,635	72,635	72,635	72,635	72,635	72,635
Cu Concentrate Payables																	
Copper payables	%						96.6%	96.6%	96.6%	96.6%	96.6%	96.6%	96.6%	96.6%	96.6%	96.6%	96.6%
Min Deduction	unit						1.5	1.5	1.5	1.5	1.5	1.2	1.4	1.1	1.2	1.4	1.4
Gold	ounces	3,694,463	95.1%				7,274	190,890	136,289	155,250	201,180	271,231	235,730	315,184	230,685	171,401	190,743
Copper	pounds	381,771,951	85.9%				727,685	9,288,322	11,862,392	14,182,890	18,287,443	26,744,253	20,882,555	31,750,103	26,057,604	20,516,816	19,501,030
Silver	ounces	9,446,983	84.6%				24,805	353,842	387,793	426,664	527,962	600,459	501,169	613,640	473,657	379,908	476,887
Dore Payables																	
Gold	ounces	1,694,151	100.0%				-	-	56,909	62,965	84,191	121,998	96,808	140,889	100,967	85,053	107,308
Silver	ounces	404,355	100.0%				-	-	15,848	16,969	21,163	25,146	19,924	25,528	19,658	17,349	23,124

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Table 22-14: Detailed Cash Flow – Mill Production Calculations (Year 11 – 29)

		Total	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	Year 28	Year 29
Mill Production																					
Total Mill Feed	tonnes	379,518,417	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	18,250,000	946,993	-	-	-	-	-	-	-
Gold	gpt	0.493	0.61	0.37	0.44	0.50	0.34	0.33	0.35	0.59	0.46	0.38	0.29	0.19	-	-	-	-	-	-	-
Copper	%	0.058	0.06	0.06	0.05	0.05	0.03	0.04	0.05	0.07	0.06	0.06	0.04	0.04	-	-	-	-	-	-	-
Silver	gpt	0.996	1.01	0.80	0.89	1.03	0.76	0.79	0.92	1.27	1.14	1.16	0.65	0.60	-	-	-	-	-	-	-
87 Pit Feed	**tonnes**	**166,084,772**	**-**	**2,600,113**	**4,226,055**	**9,390,568**	**12,429,578**	**13,543,972**	**11,616,256**	**13,243,469**	**5,284,808**	**894,838**	**3,431,153**	**276,804**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Percentage of feed tonnes	%	43.8%	0.0%	14.2%	23.2%	51.5%	68.1%	74.2%	63.7%	72.6%	29.0%	4.9%	18.8%	29.2%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Gold	gpt	0.55	-	0.32	0.42	0.44	0.39	0.37	0.45	0.67	0.74	0.19	0.19	0.19	-	-	-	-	-	-	-
Copper	%	0.06	-	0.05	0.05	0.04	0.03	0.04	0.06	0.08	0.08	0.03	0.03	0.03	-	-	-	-	-	-	-
Silver	gpt	1.119	-	0.89	1.05	0.99	0.82	0.86	1.11	1.30	1.32	0.62	0.62	0.62	-	-	-	-	-	-	-
J Pit	**tonnes**	**125,164,622**	**18,250,000**	**14,716,732**	**13,608,403**	**8,859,432**	**4,752,619**	**3,701,066**	**5,217,122**	**498,969**	**2,074,211**	**1,703,147**	**6,530,523**	**526,842**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Percentage of feed tonnes	%	33.0%	100.0%	80.6%	74.6%	48.5%	26.0%	20.3%	28.6%	2.7%	11.4%	9.3%	35.8%	55.6%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Gold	gpt	0.44	0.61	0.38	0.45	0.56	0.24	0.18	0.18	0.18	0.18	0.18	0.18	0.18	-	-	-	-	-	-	-
Copper	%	0.06	0.06	0.06	0.06	0.06	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	-	-	-	-	-	-	-
Silver	gpt	0.883	1.01	0.79	0.85	1.08	0.63	0.58	0.58	0.58	0.58	0.58	0.58	0.58	-	-	-	-	-	-	-
SW Pit	**tonnes**	**51,864,267**	**-**	**933,155**	**415,542**	**-**	**1,067,803**	**1,004,962**	**1,416,621**	**135,487**	**563,217**	**462,461**	**1,773,253**	**143,055**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Percentage of feed tonnes	%	13.7%	0.0%	5.1%	2.3%	0.0%	5.9%	5.5%	7.8%	0.7%	3.1%	2.5%	9.7%	15.1%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Gold	gpt	0.49	-	0.28	0.28	-	0.23	0.23	0.23	0.23	0.23	0.23	0.23	0.23	-	-	-	-	-	-	-
Copper	%	0.05	-	0.04	0.04	-	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	-	-	-	-	-	-	-
Silver	gpt	0.758	-	0.65	0.65	-	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	-	-	-	-	-	-	-
X22 Pit	**tonnes**	**36,404,756**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**4,372,075**	**10,327,764**	**15,189,554**	**6,515,071**	**292**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Percentage of feed tonnes	%	9.6%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	24.0%	56.6%	83.2%	35.7%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Gold	gpt	0.41	-	-	-	-	-	-	-	0.39	0.38	0.41	0.47	0.20	-	-	-	-	-	-	-
Copper	%	0.06	-	-	-	-	-	-	-	0.05	0.06	0.06	0.05	0.01	-	-	-	-	-	-	-
Silver	gpt	1.16	-	-	-	-	-	-	-	1.29	1.20	1.27	0.74	0.32	-	-	-	-	-	-	-
Combined Tail Assay - Weighted																					
Gold			0.036	0.036	0.035	0.032	0.033	0.032	0.034	0.030	0.031	0.030	0.034	0.038	-	-	-	-	-	-	-
Copper			0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.004	0.003	0.004	0.005	-	-	-	-	-	-	-
Silver			0.119	0.106	0.098	0.072	0.057	0.052	0.061	0.052	0.092	0.113	0.102	0.094	-	-	-	-	-	-	-
Combined Recovery																					
Gold			94.1%	90.1%	92.1%	93.5%	90.4%	90.1%	90.5%	95.0%	93.3%	92.1%	88.2%	80.4%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Copper			92.1%	91.5%	90.9%	90.0%	85.6%	88.0%	90.4%	93.6%	93.5%	94.2%	89.5%	86.5%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Silver			88.3%	86.7%	89.0%	93.0%	92.4%	93.5%	93.3%	95.9%	92.0%	90.2%	84.2%	84.3%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Recovered Metals																					
Gold		New																			
Total Gold Recovered	ounces	5,580,993	335,225	193,120	237,087	272,712	181,240	173,072	187,827	328,946	249,200	203,974	151,195	4,726	-	-	-	-	-	-	-
Gravity Percentage	%	28.3%	34.9%	33.7%	33.8%	33.3%	31.9%	31.8%	31.8%	32.5%	33.5%	34.4%	33.0%	31.9%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Gold - Dore	ounces	1,694,574	116,958	65,136	80,217	90,659	57,822	54,960	59,696	106,795	83,575	70,133	49,815	1,506	-	-	-	-	-	-	-
Gold - Concentrate	ounces	3,886,418	218,266	127,984	156,870	182,053	123,418	118,112	128,131	222,152	165,625	133,842	101,380	3,220	-	-	-	-	-	-	-
Copper																					
Copper Recovered	tonnes	201,622	10,491	9,614	8,994	8,146	5,391	6,623	8,518	12,193	10,429	10,266	6,648	297	-	-	-	-	-	-	-
	pounds	444,495,765	23,127,893	21,194,432	19,827,588	17,959,732	11,886,021	14,601,347	18,779,593	26,881,316	22,992,103	22,632,789	14,656,945	655,759	-	-	-	-	-	-	-
Silver																					
Total Silver Recovered	ounces	11,168,597	525,956	405,514	464,954	562,893	410,816	433,868	502,988	714,739	617,416	613,834	319,324	15,310	-	-	-	-	-	-	-
Gravity Percentage	%	3.4%	4.1%	4.0%	4.0%	3.9%	3.8%	3.8%	3.8%	3.9%	4.0%	4.0%	3.9%	3.8%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Silver - Dore	ounces	404,355	21,558	16,137	18,576	22,203	15,639	16,459	19,071	27,660	24,485	24,836	12,443	579	-	-	-	-	-	-	-
Silver - Concentrate	ounces	10,764,242	504,398	389,377	446,378	540,690	395,177	417,409	483,917	687,079	592,931	588,998	306,881	14,731	-	-	-	-	-	-	-
Total																					
Gold - Dore	ounces	1,694,574	116,958	65,136	80,217	90,659	57,822	54,960	59,696	106,795	83,575	70,133	49,815	1,506	-	-	-	-	-	-	-
Gold - Flotation	ounces	3,886,418	218,266	127,984	156,870	182,053	123,418	118,112	128,131	222,152	165,625	133,842	101,380	3,220	-	-	-	-	-	-	-
Gold Total	*ounces*	*5,580,993*	*335,225*	*193,120*	*237,087*	*272,712*	*181,240*	*173,072*	*187,827*	*328,946*	*249,200*	*203,974*	*151,195*	*4,726*	-	-	-	-	-	-	-
Copper	pounds	444,495,765	23,127,893	21,194,432	19,827,588	17,959,732	11,886,021	14,601,347	18,779,593	26,881,316	22,992,103	22,632,789	14,656,945	655,759	-	-	-	-	-	-	-
Silver - Dore	ounces	404,355	21,558	16,137	18,576	22,203	15,639	16,459	19,071	27,660	24,485	24,836	12,443	579	-	-	-	-	-	-	-
Silver - Concentrate	ounces	10,764,242	504,398	389,377	446,378	540,690	395,177	417,409	483,917	687,079	592,931	588,998	306,881	14,731	-	-	-	-	-	-	-
Silver Total	*ounces*	*11,168,597*	*525,956*	*405,514*	*464,954*	*562,893*	*410,816*	*433,868*	*502,988*	*714,739*	*617,416*	*613,834*	*319,324*	*15,310*	-	-	-	-	-	-	-
Concentrate Yield		%	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	-	-	-	-	-	-	-
Cu Concentrate (DMT) - Grades	DMT	1,518,074	73,000	73,000	73,000	73,000	73,000	73,000	73,000	73,000	73,000	73,000	73,000	3,788	-	-	-	-	-	-	-
Gold	g/DMT	79.6	93.0	54.5	66.8	77.6	52.6	50.3	54.6	94.7	70.6	57.0	43.2	26.4	-	-	-	-	-	-	-
Copper	%	13.3%	14.4%	13.2%	12.3%	11.2%	7.4%	9.1%	11.7%	16.7%	14.3%	14.1%	9.1%	7.9%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Silver	g/DMT	228.8	214.9	165.9	190.2	230.4	168.4	177.8	206.2	292.7	252.6	251.0	130.8	121.0	-	-	-	-	-	-	-
Cu Concentrate (WMT)	WMT	1,650,080	79,348	79,348	79,348	79,348	79,348	79,348	79,348	79,348	79,348	79,348	79,348	4,117	-	-	-	-	-	-	-
Cu Concentrate Delivered to Smelter (less losses)	DMT	1,510,483	72,635	72,635	72,635	72,635	72,635	72,635	72,635	72,635	72,635	72,635	72,635	3,769	-	-	-	-	-	-	-
Cu Concentrate Payables																					
Copper payables	%		96.6%	96.6%	96.6%	96.6%	96.6%	96.6%	96.6%	96.6%	96.6%	96.6%	96.6%	96.6%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Min Deduction	unit		1.4	1.5	1.5	1.5	1.5	1.5	1.5	1.3	1.4	1.4	1.5	1.5	-	-	-	-	-	-	-
Gold	ounces	3,694,463	208,254	120,220	148,386	172,943	115,768	110,594	120,363	212,043	156,923	125,932	94,278	2,902	-	-	-	-	-	-	-
Copper	pounds	381,771,951	20,064,406	18,051,316	16,737,534	14,942,191	9,104,269	11,714,183	15,730,224	23,826,804	19,933,887	19,588,522	11,767,623	509,900	-	-	-	-	-	-	-
Silver	ounces	9,446,983	441,516	330,503	385,518	476,543	336,101	357,558	421,748	617,830	526,963	523,167	250,882	11,866	-	-	-	-	-	-	-
Dore Payables																					
Gold	ounces	1,694,151	116,929	65,120	80,197	90,636	57,807	54,947	59,681	106,768	83,554	70,115	49,803	1,506	-	-	-	-	-	-	-
Silver	ounces	404,355	21,558	16,137	18,576	22,203	15,639	16,459	19,071	27,660	24,485	24,836	12,443	579	-	-	-	-	-	-	-

Table 22-15: Detailed Cash Flow – Cost and Revenue Calculations (Year -3 – Year 10)

		Total			Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Mine Production					Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Open Pit		379,518,417															
Mill Feed																	
Feed to Mill	tonnes	310,275,879			-	-	714,286	6,936,336	13,946,020	16,323,867	18,169,880	18,250,000	17,539,633	18,250,000	17,714,814	18,250,000	18,250,000
Feed to Stockpile	tonnes	69,242,538			-	628,684	6,155,276	5,820,662	2,845,968	168,402	2,681,424	5,461,920	3,806,666	8,036,707	3,973,895	5,906,577	8,051,738
Stockpile to Mill	tonnes	69,242,538			-	-	-	5,920,806	4,303,976	1,926,133	80,120	-	710,367	-	535,186	-	-
Waste	tonnes	1,170,910,012			-	9,718,464	46,633,594	67,243,006	63,224,526	63,507,731	59,148,696	62,280,993	65,504,914	60,564,506	65,162,503	55,843,423	53,698,262
Total Material	tonnes	1,619,670,967			-	10,347,148	53,503,156	85,920,810	84,320,490	81,926,133	80,080,120	85,992,913	87,561,580	86,851,213	87,386,398	80,000,000	80,000,000
Strip Ratio		3.09			-	-	65.29	5.23	3.46	3.48	3.24	3.41	3.59	3.32	3.57	3.06	2.94
Underground																	
Feed to Mill	tonnes	-													-	-	-
Operating Cost			$/t feed	$/t mined													
Open Pit Mining	dollars	4,394,463,668	11.58	2.96	Prestripping capitalized			211,872,554	232,562,093	232,905,906	238,454,257	228,589,135	230,164,829	235,403,732	232,660,885	214,946,962	206,319,455
Underground Mining	dollars	0	0.00														
Processing	dollars	2,139,398,159	5.64		-	-	3,971,430	71,485,710	102,929,977	102,930,000	102,930,000	102,930,000	102,930,000	102,930,000	102,930,000	102,930,000	102,930,000
G&A	dollars	430,496,997	1.13		Construction Indirects			21,216,713	21,630,332	21,991,630	22,358,135	22,636,443	20,335,182	20,126,656	19,812,528	19,807,890	19,436,941
Concentrate Trucking	dollars	119,630,805	0.32		-	-	225,155	4,052,795	5,752,716	5,752,717	5,752,717	5,752,717	5,752,717	5,752,717	5,752,717	5,752,717	5,752,717
Port Costs	dollars	33,001,601	0.09		-	-	62,112	1,118,012	1,586,956	1,586,957	1,586,957	1,586,957	1,586,957	1,586,957	1,586,957	1,586,957	1,586,957
Shipping to Smelter	dollars	107,255,205	0.28		-	-	201,863	3,633,540	5,157,608	5,157,609	5,157,609	5,157,609	5,157,609	5,157,609	5,157,609	5,157,609	5,157,609
Subtotal Operating	dollars	7,224,246,436			-	-	4,460,561	313,379,324	369,619,681	370,324,819	376,239,674	366,652,860	365,927,293	370,957,671	367,900,696	350,182,135	341,183,678
Capital Cost																	
Open Pit Mining	dollars	357,606,918			0	82,235,404	176,109,478	10,359,817	14,987,640	3,150,771	3,457,636	5,203,887	5,075,445	3,243,111	2,899,821	3,435,756	1,494,305
Underground Mining	dollars	0			0	0	0	0	0	0	0	0	0	0	0	0	0
Processing	dollars	458,100,024			3,439,438	167,838,147	271,678,323	8,010,154	4,864,141	0	2,269,822	0	0	0	0	0	0
Infrastructure	dollars	128,000,502			9,755,351	25,812,774	64,726,782	9,672,739	3,751,334	741,844	1,792,519	1,247,168	1,253,485	721,101	1,137,932	392,724	538,058
Environment Costs	dollars	78,178,438			0	0	10,745,313	2,149,063	2,149,063	2,149,063	2,149,063	2,149,063	2,149,063	2,149,063	2,149,063	2,149,063	2,149,063
Indirect	dollars	223,422,556			25,049,807	84,936,764	62,982,326	30,915,793	5,356,627	0	684	5,555,556	0	0	0	0	0
Contingency	dollars	105,875,790			3,947,293	35,935,709	49,420,601	5,307,102	2,723,542	82,782	422,153	1,264,440	152,153	78,678	120,769	39,272	59,619
Subtotal Capital	dollars	1,351,184,228			42,191,889	396,758,798	635,662,822	66,414,667	33,832,346	6,124,460	10,091,876	15,420,113	8,630,146	6,191,953	6,307,584	6,016,815	4,241,044
Revenue (after smelting, refining, payables, etc)			$USD/DMT														
Copper Concentrate																	
Gold	dollars	7,278,091,326	$ 4,818		-	-	14,330,280	376,053,384	268,488,500	305,842,987	396,323,936	534,326,031	464,388,185	620,911,877	454,448,685	337,659,623	375,762,756
Copper	dollars	1,400,472,748	$ 927		-	-	2,672,193	32,962,304	41,536,390	50,755,727	67,063,114	100,662,021	77,373,495	120,550,264	97,933,964	75,920,416	71,884,698
Silver	dollars	212,557,107	$ 141		-	-	558,117	7,961,454	8,725,350	9,599,943	11,879,139	13,510,327	11,276,314	13,806,899	10,657,283	8,547,935	10,729,953
Subtotal Copper Concentrate Revenue	dollars	8,891,121,181	$ 5,886		-	-	17,560,590	416,977,142	318,750,240	366,198,657	475,266,189	648,498,378	553,037,994	755,269,039	563,039,932	422,127,973	458,377,407
Dore																	
Gold	dollars	3,344,253,667			-	-	-	-	112,339,227	124,292,422	166,192,944	240,824,389	191,098,118	278,114,101	199,308,072	167,895,161	211,826,171
Silver	dollars	9,037,336			-	-	-	-	354,197	379,258	473,003	562,013	445,298	570,547	439,358	387,759	516,823
Royalties																	
Gold	dollars	106,223,450			-	-	143,303	3,760,534	3,808,277	4,301,354	5,625,169	7,751,504	6,554,863	8,990,260	6,537,568	5,055,548	5,875,889
Copper	dollars	14,004,727			-	-	26,722	329,623	415,364	507,557	670,631	1,006,620	773,735	1,205,503	979,340	759,204	718,847
Silver	dollars	2,215,944			-	-	5,581	79,615	90,795	99,792	123,521	140,723	117,216	143,774	110,966	89,357	112,468
Total Royalties	dollars	122,444,122			-	-	175,606	4,169,771	4,314,437	4,908,703	6,419,321	8,898,848	7,445,814	10,339,537	7,627,874	5,904,109	6,707,204
Total Project Revenue	**dollars**	**$ 12,121,968,062**			**-**	**-**	**17,384,984**	**412,807,371**	**427,129,228**	**485,961,635**	**635,512,815**	**880,985,932**	**737,135,595**	**1,023,614,151**	**755,159,488**	**584,506,784**	**664,013,197**
Pre-Tax Cashflow																	
Operating Cost	dollars	7,224,246,436			-	-	4,460,561	313,379,324	369,619,681	370,324,819	376,239,674	366,652,860	365,927,293	370,957,671	367,900,696	350,182,135	341,183,678
Capital Cost	dollars	1,351,184,228			42,191,889	396,758,798	635,662,822	66,414,667	33,832,346	6,124,460	10,091,876	15,420,113	8,630,146	6,191,953	6,307,584	6,016,815	4,241,044
Revenue	dollars	12,121,968,062			-	-	17,384,984	412,807,371	427,129,228	485,961,635	635,512,815	880,985,932	737,135,595	1,023,614,151	755,159,488	584,506,784	664,013,197
Pre-Tax Cashflow	dollars	3,546,537,398			-42,191,889	-396,758,798	-622,738,399	33,013,380	23,677,200	109,512,356	249,181,265	498,912,960	362,578,156	646,464,527	380,951,208	228,307,834	318,588,475
Pre-Tax Cumulative Cashflow	dollars				-42,191,889	-438,950,687	-1,061,689,086	-1,028,675,706	-1,004,998,507	-895,486,151	-646,304,886	-147,391,927	215,186,229	861,650,757	1,242,601,965	1,470,909,798	1,789,498,273
Post-Tax Cashflow																	
Quebec Taxes	dollars	909,721,086			-	-	17,561	416,977	431,444	490,870	5,475,635	32,750,602	75,804,585	171,933,343	91,512,415	49,573,340	75,289,134
Federal Taxes	dollars	432,258,881			-	-	-	-	-	-	-	-	30,035,614	74,210,137	44,487,684	26,877,644	38,449,429
Total Tax	dollars	1,341,979,967	37.8%		-	-	17,561	416,977	431,444	490,870	5,475,635	32,750,602	105,840,200	246,143,480	136,000,098	76,450,984	113,738,563
Post-Tax Cashflow	dollars	2,204,557,431			-42,191,889	-396,758,798	-622,755,960	32,596,402	23,245,756	109,021,485	243,705,630	466,162,358	256,737,957	400,321,047	244,951,110	151,856,850	204,849,911
Post-Tax Cumulative Cashflow	dollars				-42,191,889	-438,950,687	-1,061,706,647	-1,029,110,244	-1,005,864,488	-896,843,003	-653,137,373	-186,975,015	69,762,942	470,083,989	715,035,098	866,891,948	1,071,741,860
		Pre-Tax	**Post Tax**														
NPV (millions) @	**5%**	$1,564	$885														
Payback Period	**years**	5.4	5.7														
Mine Life	**years**	21.1															
IRR	**%**	18.1%	14.0%														

Table 22-16: Detailed Cash Flow – Cost and Revenue Calculations (Year 11 – Year 29)

		Total		Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	Year 28	Year 29
Mine Production				Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	Year 28	Year 29
Open Pit		379,518,417																				
Mill Feed																						
Feed to Mill	tonnes	310,275,879		18,250,000	9,739,537	14,449,578	18,238,400	11,195,169	11,603,437	8,877,968	17,353,495	14,522,777	15,189,230	6,511,453	-	-	-	-	-	-	-	-
Feed to Stockpile	tonnes	69,242,538		7,888,656	3,230,642	3,474,875	1,053,338	37,833	9,068	3,584	2,389	1,667	2,566	-	-	-	-	-	-	-	-	-
Stockpile to Mill	tonnes	69,242,538		-	8,510,463	3,800,422	11,600	7,054,831	6,646,563	9,372,032	896,505	3,727,223	3,060,770	11,738,547	946,993	-	-	-	-	-	-	-
Waste	tonnes	1,170,910,012		52,875,037	62,029,822	60,228,137	56,061,746	50,904,580	49,185,784	36,570,132	51,460,491	45,889,201	24,809,130	8,365,337	-	-	-	-	-	-	-	-
Total Material	tonnes	1,619,670,967		79,013,693	83,510,463	81,953,011	75,365,085	69,192,413	67,444,852	54,823,715	69,712,880	64,140,868	43,061,697	26,615,337	946,993	-	-	-	-	-	-	-
Strip Ratio		3.09		2.90	3.40	3.30	3.07	2.79	2.70	2.00	2.82	2.51	1.36	0.46	-	-	-	-	-	-	-	-
Underground																						
Feed to Mill	tonnes	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Operating Cost			$/t feed																			
Open Pit Mining	dollars	4,394,463,668	11.58	209,124,412	198,085,987	239,968,728	236,603,438	228,552,693	257,700,025	214,164,267	210,719,041	156,937,455	116,880,001	58,269,304	3,578,511	0	0	0	0	0	0	0
Underground Mining	dollars	0	0.00																			
Processing	dollars	2,139,398,159	5.64	102,930,000	102,930,000	102,930,000	102,930,000	102,930,000	102,930,000	102,930,000	102,930,000	102,930,000	102,930,000	102,930,000	5,341,042	-	-	-	-	-	-	-
G&A	dollars	430,496,997	1.13	19,923,921	19,583,459	19,882,650	19,904,681	18,796,615	19,328,835	18,713,131	19,138,759	18,201,429	17,499,215	15,926,032	14,245,820	0	0	0	0	0	0	0
Concentrate Trucking	dollars	119,630,805	0.32	5,752,717	5,752,717	5,752,717	5,752,717	5,752,717	5,752,717	5,752,717	5,752,717	5,752,717	5,752,717	5,752,717	298,509	-	-	-	-	-	-	-
Port Costs	dollars	33,001,601	0.09	1,586,957	1,586,957	1,586,957	1,586,957	1,586,957	1,586,957	1,586,957	1,586,957	1,586,957	1,586,957	1,586,957	82,347	-	-	-	-	-	-	-
Shipping to Smelter	dollars	107,255,205	0.28	5,157,609	5,157,609	5,157,609	5,157,609	5,157,609	5,157,609	5,157,609	5,157,609	5,157,609	5,157,609	5,157,609	267,629	-	-	-	-	-	-	-
Subtotal Operating	dollars	7,224,246,436		344,475,616	333,096,729	375,278,660	371,935,401	362,776,591	392,456,143	348,304,681	345,285,082	290,566,167	249,806,498	189,622,618	23,813,858	-	-	-	-	-	-	-
Capital Cost																						
Open Pit Mining	dollars	357,606,918		5,138,573	9,224,482	11,764,221	6,505,587	3,534,544	1,289,576	5,388,159	1,804,776	897,661	406,267	0	0	0	0	0	0	0	0	0
Underground Mining	dollars	0		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Processing	dollars	458,100,024		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Infrastructure	dollars	128,000,502		538,058	538,058	538,058	538,058	538,058	538,058	538,058	538,058	538,058	538,058	538,058	538,058	0	0	0	0	0	0	0
Environment Costs	dollars	78,178,438		2,149,063	2,149,063	2,149,063	2,149,063	2,149,063	2,149,063	2,149,063	2,149,063	2,149,063	2,149,063	2,149,063	2,149,063	-1,408,750	2,875,000	2,875,000	2,875,000	2,875,000	2,875,000	7,187,500
Indirect	dollars	223,422,556		0	0	0	0	0	0	0	0	0	0	0	0	2,156,250	862,500	862,500	862,500	862,500	862,500	2,156,250
Contingency	dollars	105,875,790		59,619	59,619	59,619	59,619	59,619	59,619	59,619	59,619	59,619	59,619	59,619	59,619	1,401,563	560,625	560,625	560,625	560,625	560,625	1,401,563
Subtotal Capital	dollars	1,351,184,228		7,885,312	11,971,221	14,510,960	9,252,326	6,281,283	4,036,315	8,134,898	4,551,515	3,644,400	3,153,007	2,746,739	2,746,739	2,149,063	4,298,125	4,298,125	4,298,125	4,298,125	4,298,125	10,745,313
Revenue (after smelting, refining, payables, etc)			$USD/DMT																			
Copper Concentrate																						
Gold	dollars	7,278,091,326	$ 4,818	410,261,218	236,833,273	292,321,097	340,696,951	228,062,030	217,869,680	237,115,772	417,724,502	309,139,162	248,085,867	185,727,749	5,717,781	-	-	-	-	-	-	-
Copper	dollars	1,400,472,748	$ 927	74,122,989	66,124,985	60,905,328	53,772,429	30,578,366	40,947,555	56,903,284	89,070,996	73,604,438	72,232,304	41,159,870	1,735,617	-	-	-	-	-	-	-
Silver	dollars	212,557,107	$ 141	9,934,102	7,436,325	8,674,153	10,722,213	7,562,266	8,045,052	9,489,333	13,901,169	11,856,673	11,771,269	5,644,848	266,993	-	-	-	-	-	-	-
Subtotal Copper Concentrate Revenue	dollars	8,891,121,181	$ 5,886	494,318,309	310,394,583	361,900,579	405,191,593	266,202,662	266,862,287	303,508,390	520,696,668	394,600,272	332,089,439	232,532,467	7,720,391	-	-	-	-	-	-	-
Dore																						
Gold	dollars	3,344,253,667		230,818,018	128,546,743	158,309,748	178,915,694	114,111,898	108,464,542	117,810,389	210,759,747	164,936,015	138,407,296	98,310,195	2,972,775	-	-	-	-	-	-	-
Silver	dollars	9,037,336		481,815	360,660	415,168	496,238	349,527	367,866	426,229	618,210	547,229	555,088	278,102	12,950	-	-	-	-	-	-	-
Royalties																						
Gold	dollars	106,223,450		6,410,792	3,653,800	4,506,308	5,196,126	3,421,739	3,263,342	3,549,262	6,284,842	4,740,752	3,864,932	2,840,379	86,906	-	-	-	-	-	-	-
Copper	dollars	14,004,727		741,230	661,250	609,053	537,724	305,784	409,476	569,033	890,710	736,044	722,323	411,599	17,356	-	-	-	-	-	-	-
Silver	dollars	2,215,944		104,159	77,970	90,893	112,185	79,118	84,129	99,156	145,194	124,039	123,264	59,230	2,799	-	-	-	-	-	-	-
Total Royalties	dollars	122,444,122		7,256,181	4,393,020	5,206,255	5,846,035	3,806,641	3,756,947	4,217,450	7,320,746	5,600,835	4,710,518	3,311,208	107,061	-	-	-	-	-	-	-
Total Project Revenue	**dollars**	**$ 12,121,968,062**		**718,361,961**	**434,908,965**	**515,419,240**	**578,757,490**	**376,857,446**	**371,937,747**	**417,527,557**	**724,753,879**	**554,482,681**	**466,341,305**	**327,809,557**	**10,599,055**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Pre-Tax Cashflow																						
Operating Cost	dollars	7,224,246,436		344,475,616	333,096,729	375,278,660	371,935,401	362,776,591	392,456,143	348,304,681	345,285,082	290,566,167	249,806,498	189,622,618	23,813,858	-	-	-	-	-	-	-
Capital Cost	dollars	1,351,184,228		7,885,312	11,971,221	14,510,960	9,252,326	6,281,283	4,036,315	8,134,898	4,551,515	3,644,400	3,153,007	2,746,739	2,746,739	2,149,063	4,298,125	4,298,125	4,298,125	4,298,125	4,298,125	10,745,313
Revenue	dollars	12,121,968,062		718,361,961	434,908,965	515,419,240	578,757,490	376,857,446	371,937,747	417,527,557	724,753,879	554,482,681	466,341,305	327,809,557	10,599,055	-	-	-	-	-	-	-
Pre-Tax Cashflow	dollars	3,546,537,398		366,001,033	89,841,015	125,629,619	197,569,763	7,799,572	-24,554,711	61,087,979	374,917,282	260,272,114	213,381,800	135,440,200	-15,961,542	-2,149,063	-4,298,125	-4,298,125	-4,298,125	-4,298,125	-4,298,125	-10,745,313
Pre-Tax Cumulative Cashflow	dollars			2,155,499,306	2,245,340,321	2,370,969,941	2,568,539,704	2,576,339,276	2,551,784,565	2,612,872,544	2,987,789,826	3,248,061,940	3,461,443,740	3,596,883,940	3,580,922,398	3,578,773,335	3,574,475,210	3,570,177,085	3,565,878,960	3,561,580,835	3,557,282,710	3,546,537,398
Post-Tax Cashflow																						
Quebec Taxes	dollars	909,721,086		90,673,090	16,791,765	26,589,415	40,384,951	593,321	375,695	5,290,002	93,337,409	60,796,025	47,926,708	23,256,092	10,706	-	-	-	-	-	-	-
Federal Taxes	dollars	432,258,881		45,141,910	11,862,860	16,787,342	20,622,153	209,535	-	3,939,830	47,042,717	33,164,491	27,487,668	11,939,867	-	-	-	-	-	-	-	-
Total Tax	dollars	1,341,979,967	37.8%	135,815,000	28,654,626	43,376,757	61,007,105	802,856	375,695	9,229,832	140,380,126	93,960,516	75,414,375	35,195,959	10,706	-	-	-	-	-	-	-
Post-Tax Cashflow	dollars	2,204,557,431		230,186,033	61,186,390	82,252,862	136,562,658	6,996,716	-24,930,405	51,858,147	234,537,156	166,311,598	137,967,425	100,244,240	-15,972,249	-2,149,063	-4,298,125	-4,298,125	-4,298,125	-4,298,125	-4,298,125	-10,745,313
Post-Tax Cumulative Cashflow	dollars			1,301,927,892	1,363,114,282	1,445,367,144	1,581,929,803	1,588,926,518	1,563,996,113	1,615,854,260	1,850,391,416	2,016,703,014	2,154,670,439	2,254,914,680	2,238,942,431	2,236,793,368	2,232,495,243	2,228,197,118	2,223,898,993	2,219,600,868	2,215,302,743	2,204,557,431
		Pre-Tax	**Post Tax**																			
NPV (millions) @	**5%**	$1,564	$885																			
Payback Period	**years**	5.4	5.7																			
Mine Life	**years**	21.1																				
IRR	**%**	18.1%	14.0%																			

23 ADJACENT PROPERTIES

There are no significant adjacent properties to Project.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Risk and Opportunity Assessment

A risk workshop session was held on November 21st and November 22nd, 2023, at the office of Lycopodium Minerals. Participation included Troilus management, consultants from AGP, WSP, and Lycopodium. The purpose was to update the existing risk and opportunity register.

24.2 Definition of Objectives

The following outlines the definition of Project objectives:

- Health and Safety: Meet or exceed safety targets. Zero 'Lost Time' incidents during construction. Design for a safe operating environment.
- Environment: Be environmentally and socially responsible. No serious environmental incidents, comply with all permit requirements.
- Community: Maintain good community relations. Maximize utilization of local available resources and involvement of the local community. Leave a positive legacy.
- Capital Cost: Target to achieve the lowest cost outcome without compromising quality and schedule. Complete the Project within the project control budget.
- Quality: Design the Project to be fit-for-purpose, easy to maintain, operator friendly and safe.
- Schedule: Meet key milestone dates.

24.3 Identification of Risks

Previously identified risks were reviewed, assessed, and the risk register updated accordingly. Any new risks that were identified were added to the register.

Further details of the identified risks, causes and effects are shown in the Risk Register included in Appendix J.

24.4 Assessment and Planning of Risks

All risks were reviewed, and new risks identified. Each risk was assessed as per the method outlined below.

24.4.1 Assessment

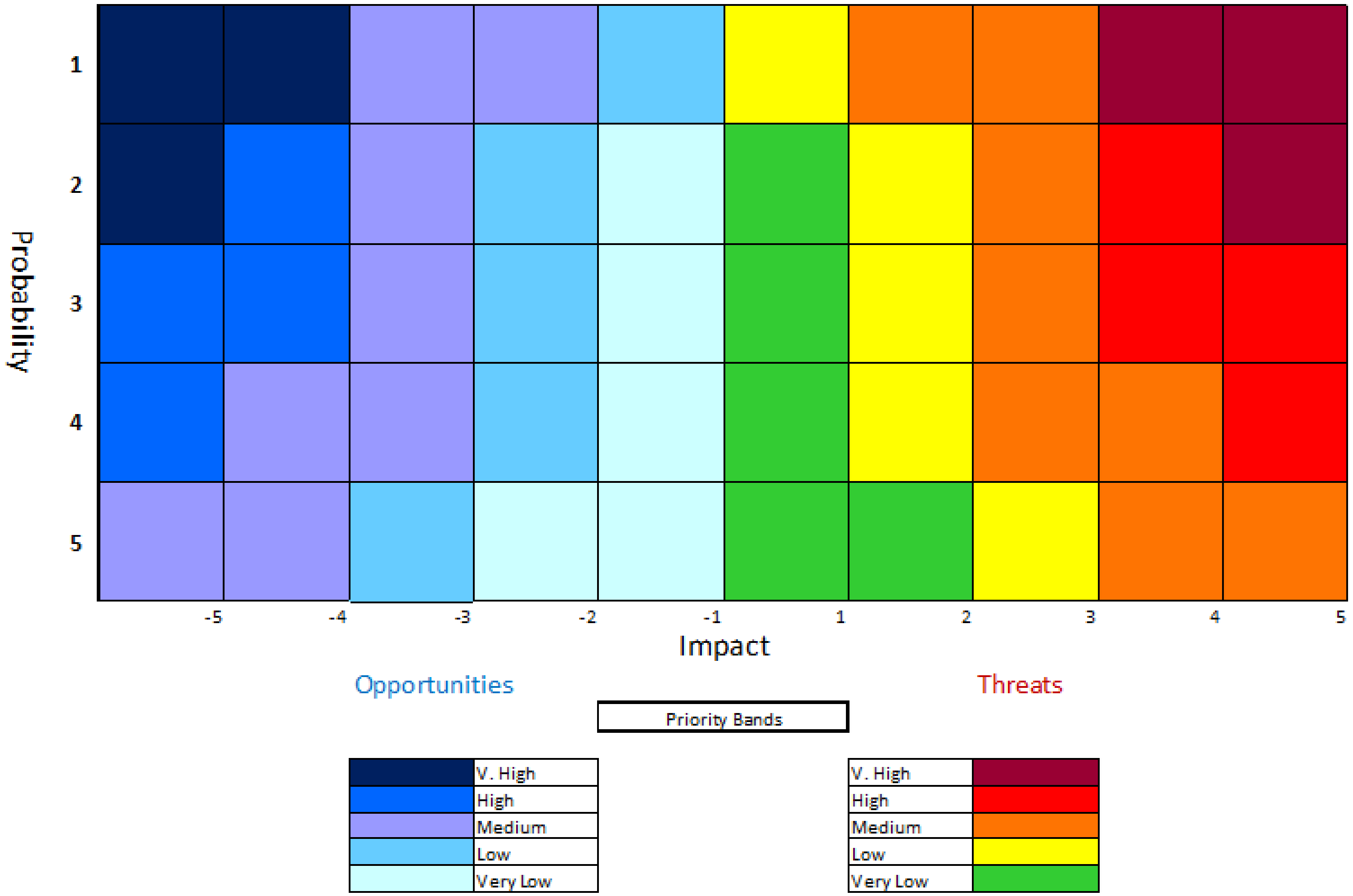
Risks that were classified as a threat, or as an opportunity, in the identification phase are further classified by qualitatively assigning the probability of occurrence of that risk and the impact of the consequence if that risk would occur.

A 5 x 10 matrix was utilized for this classification, assigning values from 1 to 5 for probability of occurrence and values from -5 to +5 for the impact (negative numbers for opportunities, positive numbers for threats). Table 24-1, Figure 24-1, and Figure 24-2 provide the guidance utilized in the Project for the classification of risks.

Table 24-1: Probability and Impact Scales

	Probability	%	Impact								Definition
			Opportunity		Cost	Time	Threat		Cost	Cost	
1	Improbable	<10	-1	Minor			1	Insignificant	<$2,000,000	< 1 month	Minor incident or inefficiency of little or no consequence. HSE no injury or health consequence - near miss or non-recordable accident
2	Could happen	10-30	-2	Significant			2	Marginal	$2,000,000 - 5,000,000	1 to 3 months	Minor incident or inefficiency that may require engineering review and is easily and predictably remediated. HSE minor injury, no health consequence- recordable case, reportable environmental incident or permitting delay (1 week)
3	As likely as not	30-50	-3	Substantial			3	Serious	$5,000,000 - 15,000,000	3 to 6 months	Moderate event or inefficiency that may need physical attention, certainly engineering review. HSE minor injuries, reversible health effects - lost time injury or occupational illness, environmental reportable incident with long term effects or permitting delays (several weeks)
4	Probable	50-70	-4	Very Substantial			4	Critical	$15,000,000 - 50,000,000	6 to 12 months	Significant event or inefficiency that can be addressed but with great effort. HSE moderate injury or health effects - serious injury or occupational disease, environmental significant reportable incident not readily remediated, permitting delay (several months), potential damage to corporate image
5	Highly probable	>70	-5	Exceptional			5	Catastrophic	>$50,000,000	> 12 months	Major uncontrolled event or inefficiency with uncertain and perhaps prohibitively costly remediation. HSE fatality or severe health effects, environmental extreme incident that cannot be remediated completely, permitting denied or delayed (greater than 1 year), damage to corporate image and share value.

Figure 24-1: Risk Matrix



Lycopodium (2023)

Figure 24-2: Risk/Opportunity Profile (Frequency)

Lycopodium (2023)

The top 10 qualified risks are summarized in Table 24-2.

24.4.2 Planning

A response strategy and action plan were developed for many of the identified risks. Table 24-3 shows these strategies and action plans for the top 10 risks.

24.5 Next Steps

Risks and opportunities must be managed by the owners identified in Table 24-4 and Table 24-5. Periodic reviews of the risk register should be held between the risk owner, risk manager, and Troilus management.

Table 24-2: Top Ten Risks

Risk ID	Risk Type	Risk Description	Cause(s)	Effect(s)	Risk Owner	Prob	Impact - Cost	Impact - Time	Priority - Current
29	Geology	Delay of condemnation drilling reduces confidence in placement of site infrastructure.	No condemnation drilling has taken place to date.	Late condemnation drilling may require significant changes to project plan and schedule as a result of rework.	JR/KF	5	5	1 yr.	VERY HIGH
38	Environmental and Permitting	Obtaining permits - Federal government permitting delay due to new process.	Federal government delays to permitting process.	Delayed project.	JL	4	5	6-12 months	VERY HIGH
13	Site Infrastructure	Run off and seepage at site are low quality at or after mine closure	Unfavourable geochemistry of waste rock or tailings.	Requirement for long term water treatment after closure.	JL	3	4	>25 yrs.	HIGH
27	Geology	Gap Zone drilling determines permitting does not allow condemning, but resource does not improve project economics.	Gap zone is economically mineralized.	Significant schedule delay and permitting impact.	KF	2	5	1 yr.	HIGH
42	Environmental and Permitting	Unfavourable geochemical results significantly affecting closure plan.	Unfavourable geochemical results.	Closure bond impact.	JL	2	5	>25 yrs.	HIGH
43	Construction - Project Development	Skilled construction labour - shortage in Québec and unions.	Not enough skilled labour during the construction phase.	Delay to the construction schedule.	IP	3	4	3 yrs.	HIGH
44	Construction - Project Development	Cost - escalation and inflation.	Market conditions.	Increase in CAPEX.	IP	3	4	3 yrs.	HIGH

48	Construction - Project Development	Road traffic accidents during construction phase.	Increase in traffic volume during construction phase.	Serious injury, loss of life, potential environmental impact, and media attention.	IP	3	5	5 yrs.	**HIGH**
49	Construction - Project Development	Natural disasters.	Disruption and delay of activities on site.	Flood, fire, pandemic, earthquake etc.	JR	2	5	5 yrs.	HIGH
52	Operations	Motor Vehicle Collisions during operations phase.	Increase in traffic volume during operations phase.	Serious injury, loss of life, potential environmental impact, and media attention.	IP	2	5	25 yrs.	HIGH
59	Operations	Natural disasters.	Disruption and delay of activities on site.	Flood, fire, pandemic, earthquake etc.	JR	2	5	25 yrs.	HIGH
61	Project Development	Project financing.	Unfavourable market conditions.	Project delays.	JR	2	5	5 yrs.	HIGH

Table 24-3: Top Ten Opportunities

Risk ID	Risk Type	Risk Description	Cause(s)	Effect(s)	Risk Owner	Prob	Impact - Cost	Impact - Time	Priority - Current
8	Plant Layout	At pit crushing of ore to reduce operation costs (haul distance, carbon emissions etc.)	Layout and sizing of plant equipment.	Reduction of operating costs dust and noise.	IP	4	-5	25 yrs.	VERY HIGH
10	Site Infrastructure	Power - Joint venture with First Nation.	First Nation willing to JV.	Potential benefit for the region's power supply.	JR	4	-5	25 yrs.	VERY HIGH
16	Mining	Phased mining improves slope stability and depressurization of J and 87	Apparent overall slope angle reduced and increased slope depressurization.	Allows deeper and steeper mining, primarily for the west walls.	IP	4	-5	25 yrs.	VERY HIGH
35	Stakeholder	Stakeholder training and education to empower the Cree Nation.	Lack of experience and training opportunities.	Empower individuals who have the skills and training to benefit themselves, their communities, and the project.	CS	5	-4	ongoing	VERY HIGH
37	Stakeholder	Early engagement with Cree Nation to explore business opportunities.	Cree Nation preparation of new businesses to partner with the project.	Cree community development and readily available services for the project.	CS	5	-5	25 yrs.	VERY HIGH
3	Met	Resource expansion and retrospectivity of the met sample of prospective new resources.	Higher grade / larger resource.	Increase in revenue and life of mine.	KF	4	-4	25 yrs.	HIGH
20	Mine Geotech	Overall geotechnical design for mine slopes - too conservative in design therefore opportunity to steepen and mine deeper during operations.	Geotech uncertainty for all ore zones. Implement learning from previous phased wall controls and performance.	Lower operating cost and increased ore production earlier.	IP	3	-5	25 yrs.	HIGH

28	Geology	Gap Zone drilling determines indicated resources that improves project economics.	Gap zone is economically mineralized.	Increase in project value.	KF	2	-5	2 yrs.	HIGH
45	Construction - Project Development	Cost - de-escalation and deflation	Market conditions.	Decrease in CAPEX.	IP	3	-4	3 yrs.	HIGH

Table 24-4: Top Ten – Action Plans (Risk & Opportunities)

Risk ID	Risk Type	Priority - Current	Risk Review Date	Response Strategy Summary	Action Plan	Action Owner	Action Review Date
29	Geology	VERY HIGH	Q4 2023	Initiate condemnation drilling.	Initiate condemnation drilling.	KF/JR	Q4 2023
38	Environmental and Permitting	VERY HIGH	2024	Engage stakeholders to rally support in favour of the project.	Engage with stakeholders. Continue engagement with federal govt.	JL	2024
13	Site Infrastructure	HIGH	H1 2024	Adaptive management of waste chemistry and contact water and develop geochemistry predicting model.	Early planning and management of waste rock placement and develop geochemistry predicting model.	JL	Ongoing
27	Geology	HIGH	H1 2024	Drill gap zone ASAP.	Drill gap zone ASAP.	KF	H1 2024
42	Environmental and Permitting	HIGH	2024	Early closure planning and characterization testwork.	Early closure planning and characterization testwork.	JL	2024
43	Construction - Project Development	HIGH	H2 2024	Earlier engagement of contractors and union.	Develop site specific labour agreement with contractors and union.	IP	H2 2024
44	Construction - Project Development	HIGH	H2 2024	Track market conditions.	Track market conditions.	IP	H2 2024
48	Construction - Project Development	HIGH	H2 2024	Drug and alcohol testing, GPS tracking, transportation, logistics plan and training.	Drug and alcohol testing, GPS tracking, transportation and logistics plan and training.	RH	H2 2024
49	Construction - Project Development	HIGH	H2 2024	Contingency planning.	Contingency planning.	JR	H2 2024
52	Operations	HIGH	H2 2024	Drug and alcohol testing, GPS tracking, bus transportation, logistics plan and training.	Drug and alcohol testing, GPS tracking, bus transportation, logistics plan and training.	IP	H2 2024
59	Operations	HIGH	H2 2024	Contingency planning.	Contingency planning.	JR	H2 2024
61	Project Development	HIGH	H2 2024	Engage project finance advisor, independent engineer.	Engage project finance advisor, independent engineer.	JR	H2 2024

Table 24-5: Top Ten – Action Plans

Risk ID	Risk Type	Priority - Current	Risk Review Date	Response Strategy Summary	Action Plan	Action Owner	Action Review Date
8	Plant Layout	VERY HIGH	H2 2024	Trade off study.	Complete trade off study.	WH	H2 2024
10	Site Infrastructure	VERY HIGH	H2 2024	Work with CNG to develop JV opportunities.	Work with CNG to develop JV opportunities.	JR/CS/CF	H2 2024
16	Mining	VERY HIGH	H2 2024	Further characterization of major structures	Further characterization of major structures during operation of the initial phases. Monitor performance during initial phases.	IP/WH/CG	Ongoing
35	Stakeholder	VERY HIGH	H1 2024	Education and training strategy.	Education and training strategy.	JL	H1 2024
37	Stakeholder	VERY HIGH	H1 2024	Early engagement and identification of potential business opportunities.	PDA committee to share potential business opportunities.	CS	Ongoing
3	Met	HIGH	H2 2024	Ensure tailings and waste storage can be expanded.	Consider flexible tailings and waste allowing for longevity of mine life.	JM/IP	H2 2024
20	Mine Geotech	HIGH	H2 2024	Collect additional data to reduce uncertainty. Site based geotechnical team to manage on site risk, monitoring.	Collect additional data to reduce uncertainty. Site based geotechnical team to manage on site risk, monitoring.	IP	Ongoing
28	Geology	HIGH	H1 2024	Drill gap zone ASAP.	Drill gap zone ASAP.	KF	H1 2024
45	Construction - Project Development	HIGH	H2 2024	Track market conditions.	Track market conditions.	IP	H2 2024

24.6 Alternative Waste Crushing, Conveying, and Stacking Process

A trade-off was completed where truck hauling of waste was compared to waste crushing, conveying, and stacking. Although truck hauling of material was determined to more economical, the waste crushing, conveying, and stacking system is presented here for reference.

24.6.1 Power

There is a 25 kV overhead power line near the northwest side of the J Zone Pit. This power line is assumed to be source of power for the Waste Crushing, Conveying and Stacking process.

The power line will be erected using wooden power poles to the crusher and then along the overland conveyor route to the southwest towards the West Waste Dump in Years Two through Ten. In Years Ten through Seventeen, the power line will be extended to the northeast towards the 87-WD dump.

The crushing plant will have a step-down transformer and two electrical rooms to supply low voltage power to the crusher.

The conveyors themselves will include 25 kV x 600 V transformers where necessary.

The power requirements for the Waste Crushing, Conveying and Stacking process are displayed in Table 24-6.

Table 24-6: Power Requirements

		Crushing	Conveying & Stacking	Total
Installed Power, kW	Years 2 through 10	1,371	14,585	15,955
	Years 10 through 17	1,371	16,338	17,709
Operating Demand, kW	Years 2 through 10	843	4,487	5,331
	Years 10 through 17	843	4,838	5,681
Peak Demand, kW	Years 2 through 10	1,097	9,942	11,039
	Years 10 through 17	1,097	9,942	11,039
Power Consumed, kWh/t	Years 2 through 10	0.18	0.98	1.17
	Years 10 through 17	0.18	1.06	1.24

24.6.2 Waste Handling

Mined waste from the northern pits (87 Pit and J Zone Pit) will be crushed in a mineral sizer and transported, using overland conveyors, to a waste dump. The waste will be forward stacked on the dump using a system of mobile Ramp, Grasshopper, Index Feed, Horizontal Index and Radial Stacker Conveyor.

Waste stacking will reduce the amount of waste that must be hauled using trucks from the pit to the waste dump.

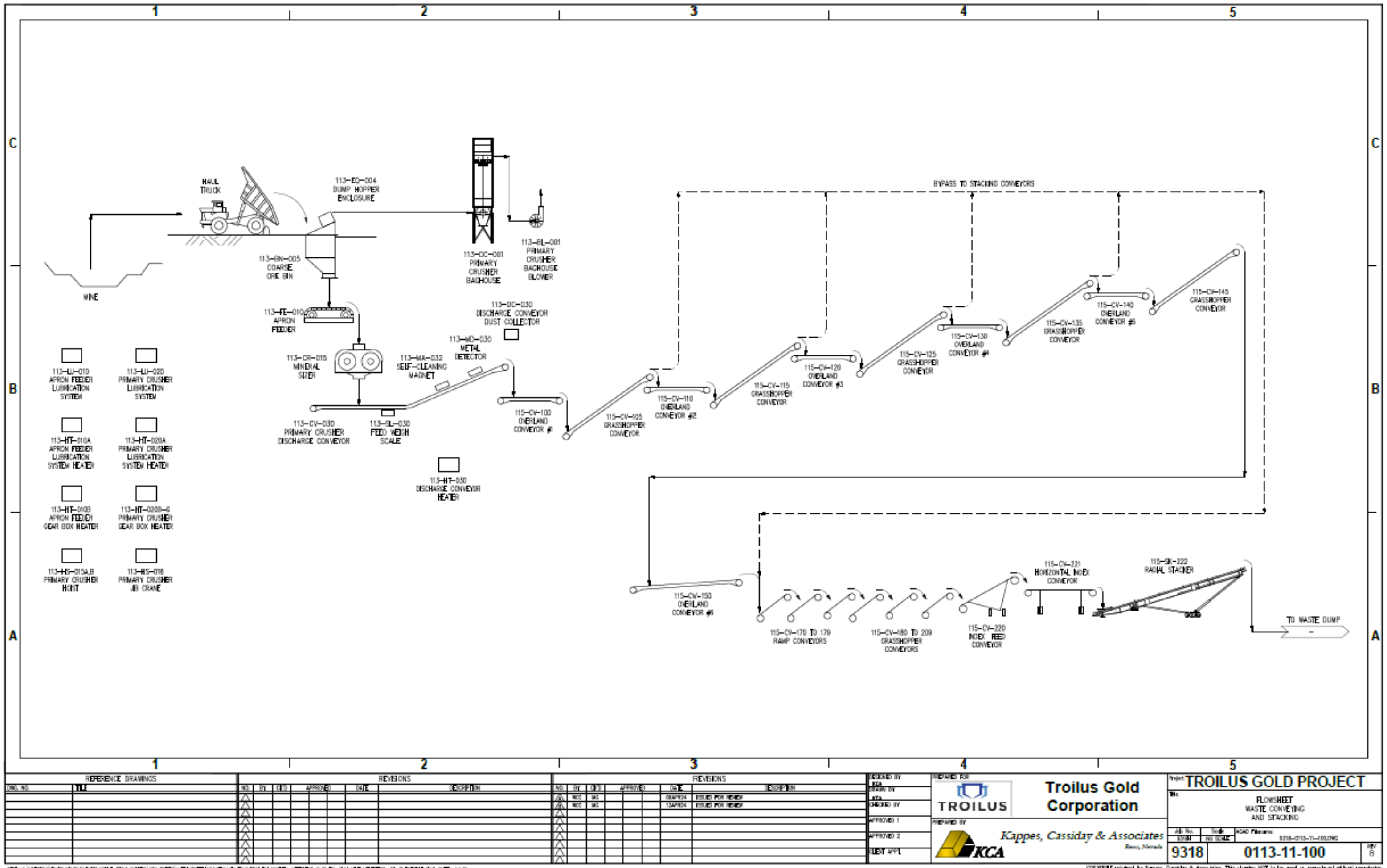
The West Waste Dump has available volume to store 153 million m^3 of waste (320 million t) and will be stacked beginning in Year 2. This will accommodate 8.4 years of conveyer stacked crushed waste.

The 87-WD, or North Waste Dump, northeast of the 87 Pit, has the volume to store 196 million m^3 (409 million t) and will be stacked in Years 10 through 17. It is assumed that the dump will stack by a combination of trucks and conveyors with 137 M tonnes truck stacked and 272 M tonnes conveyor stacked.

The waste stacking concept will be to first conveyor stack waste in the West Waste Dump, then shift to dumping in the 87-WD. Overland and stacking conveyors can be moved to the north to reduce costs.

A flowsheet for waste stacking is shown below Figure 24-3 below.

Figure 24-3: Waste Stacking Flowsheet



Source: KCA (2024)

24.6.3 Design Criteria

The design criteria for the Troilus Waste Stacking project is summarized in Table 24-7.

Table 24-7: Design Criteria

	Description	Value and Units	
Waste Crushing and Conveying	Year 2 to 17		
Conveyor Stacked Dump Location			
	Years 2 through 10	West Waste Dump	
	Years 10 through 17	North Dump, 87-WD	
ROM Bulk Density	at mine	2.09	dry mt/m^3
Bulk Density	after crushing	1.6	dry mt/m^3
Stacked Waste Bulk Density	at Dump	2.09	dry mt/m^3
Run of Mine Feed Size, F100	Fragmentation Study	295	mm
Run of Mine Feed Size, F80	Fragmentation Study	262	mm
Waste Stacking Method	Forward Stacking		
Lift Height		20	m
Overall Side Slope	2.5 H: 1 V	22	degrees
Angle of Repose		38	degrees
Abrasion Index	A_i	0.29	g
Crushing Work Index	CWi	22.5	kWh/tonne
Rod Mill Work Index	RW_i	16.5	kWh/tonne
Ball Mill Work Index	BW_i	13.8	kWh/tonne
SMC	Mih	22	kWh/tonne
SMC	Axb	26	
Mining Availability		95%	
Primary Crushing Availability, %		87.4%	
Conveying and Stacking Availability, %		89.8%	
Total Availability		74.6%	
Primary Crushing Throughput	Nominal	109,589	dry mtpd
	Nominal	4,566	dry mtph
	Design	6,122	dry mtph
Conveyor Excess Capacity		20%	
Conveyor Covers	Yes		
Nominal	Includes All Above	6,122	dry mtph
Design		7,347	dry mtph

The following values in the design criteria are conservative and the design requirements could be reduced by further evaluation:

Bulk Density after crushing	**1.6**	**dry mt/m³**
Conveyor Excess Capacity	**20%**	

24.6.4 Primary Crusher

ROM waste will be trucked from the mine in 240 t capacity haul trucks and direct dumped in a 250 m³ (523 t) dump hopper. The total volume of the hopper will accommodate 2.2 trucks. The hopper includes an enclosure exhausted to a baghouse to control dust. The enclosure will be fitted with curtains to aid in dust control.

A D10 apron feeder is a variable speed feeder to meter waste into an MMD 1000 Series 4 Mineral Sizer.

Primary crusher product is conveyed away on a 72-inch conveyor. The conveyor includes a self-cleaning magnet and metal detector.

A dust collector is installed on the discharge of the Crusher Discharge Conveyor.

Ancillary Facilities

The crusher supply includes a Control Room, a Maintenance Area to repair the sizer and two Electrical Rooms located on the ground level.

24.6.5 Overland Conveying

Years 2 through 10

Primary crushed waste is transported on Overland Conveyor #1, a 1,570 m long, 1,524 mm wide (60-inch) conveyor with a net change in elevation of -5 m.

Following Overland Conveyor #1 the waste is transferred to a series of alternating grasshopper and overland conveyors. Each overland conveyor discharges onto a grasshopper which can be positioned to feed waste to the next overland in series or to the active waste stacking cell. There are a total of six overland conveyors and five grasshopper conveyors corresponding to the waste stacking access ramps.

The overland conveyors are shown in Table 24-8:

Table 24-8: Overland Conveyor List

Conveyor		Lenth m	Width mm	Installed Power hp
115-CV-100	Crushed Product Overland 1	1,570	1,524	2 x 800
115-CV-110	Crushed Product Overland 2	307	1,524	2 x 250
115-CV-120	Crushed Product Overland 3	286	1,524	2 x 250
115-CV-130	Crushed Product Overland 4	250	1,524	2 x 200
115-CV-140	Crushed Product Overland 5	274	1,524	2 x 250
115-CV-150	Crushed Product Overland 6	285	1,524	2 x 250

The grasshopper conveyors are shown in Table 24-9:

Table 24-9: Grasshoppers in Overland Conveyor System

Conveyor		Lenth m	Width mm	Installed Power hp
115-CV-100	Grasshopper	46	1,829	2x125
115-CV-110	Grasshopper	46	1,829	2x125
115-CV-120	Grasshopper	46	1,829	2x125
115-CV-130	Grasshopper	46	1,829	2x125
115-CV-140	Grasshopper	46	1,829	2x125

Years 10 through 17

Primary crushed waste is transported on Overland Conveyor #1, a 245 m long, 1,524 mm wide (60-inch) conveyor with a net change in elevation of -5 m.

Following Overland Conveyor #1 the waste is transferred to a series of five additional overland conveyors (Overland Conveyors #2 through 6).

Overland Conveyor #6 discharges to a series of alternating grasshopper and overland conveyors. Each grasshopper can be positioned to feed waste to the next overland in series or to the active waste stacking cell. There are a total of ten overland conveyors and four grasshopper conveyors corresponding to the waste stacking access ramps.

The overland conveyors are shown in Table 24-10:

Table 24-10: Overland Conveyor List

Conveyor		Lenth m	Width mm	Installed Power hp
115-CV-200	Crushed Product Overland 1	245	1,524	2 x 250
115-CV-210	Crushed Product Overland 2	330	1,524	2 x 300
115-CV-220	Crushed Product Overland 3	350	1,524	2 x 300
115-CV-100	Crushed Product Overland 4	1420	1,524	2 x 800
115-CV-240	Crushed Product Overland 5	440	1,524	2 x 250
115-CV-110	Crushed Product Overland 6	307	1,524	2 x 450
115-CV-120	Crushed Product Overland 7	286	1,524	2 x 250
115-CV-130	Crushed Product Overland 8	250	1,524	2 x 250
115-CV-140	Crushed Product Overland 9	274	1,524	2 x 250
115-CV-150	Crushed Product Overland 10	285	1,524	2 x 250

The grasshopper conveyors are shown in Table 24-11:

Table 24-11: Grasshoppers in Overland Conveyor System

Conveyor		Lenth m	Width mm	Installed Power hp
115-CV-115	Grasshopper	46	1,829	2x125
115-CV-125	Grasshopper	46	1,829	2x125
115-CV-135	Grasshopper	46	1,829	2x125
115-CV-145	Grasshopper	46	1,829	2x125

24.6.6 Waste Stacking

West Waste Dump

The West Waste Dump will be stacked in 20 m lifts from the northeast to the southwest end.

A ramp and platform, 20 m tall, will be built by haul truck and a dozer on the west side of Cell #1. The Radial Stacker will be positioned on the platform and fed by Ramp Conveyors. The Radial Stacker will be used to build the platform out until it is large enough to accommodate both the Horizontal Index Conveyor (HIC) and the Radial Stacker. Once both the HIC and Radial Stacker are on the platform, they will be pushed forward by adding Grasshopper Conveyors, forward stacking the length of Cell #1 towards the Tailings Storage Facility (TSF) to the east.

The Radial Stacker will be turned to the south when Cell #1 is fill and a platform will be built into the TSF side of Cell #2. Once there is a platform large enough for the HIC and Radial Stacker, the Cell #2 will be forward stacked towards the west. Grasshoppers will be removed from Cell #1 as the stacked face advances to the west.

At the west end of the Cell #2, the Radial Stacker will be turned to the south to build a ramp and platform in Cell #3. The stacking conveyors will be fed from the grasshopper on the end of Overland Conveyor 2.

All conveyors will include a powered bin vent on their discharge chutes to reduce dust.

The above process will be repeated as the dump grows to the south.

The stacking process is documented in Table 24-12 and Table 24-13 below, as well as Figure 24-4 and Figure 24-5.

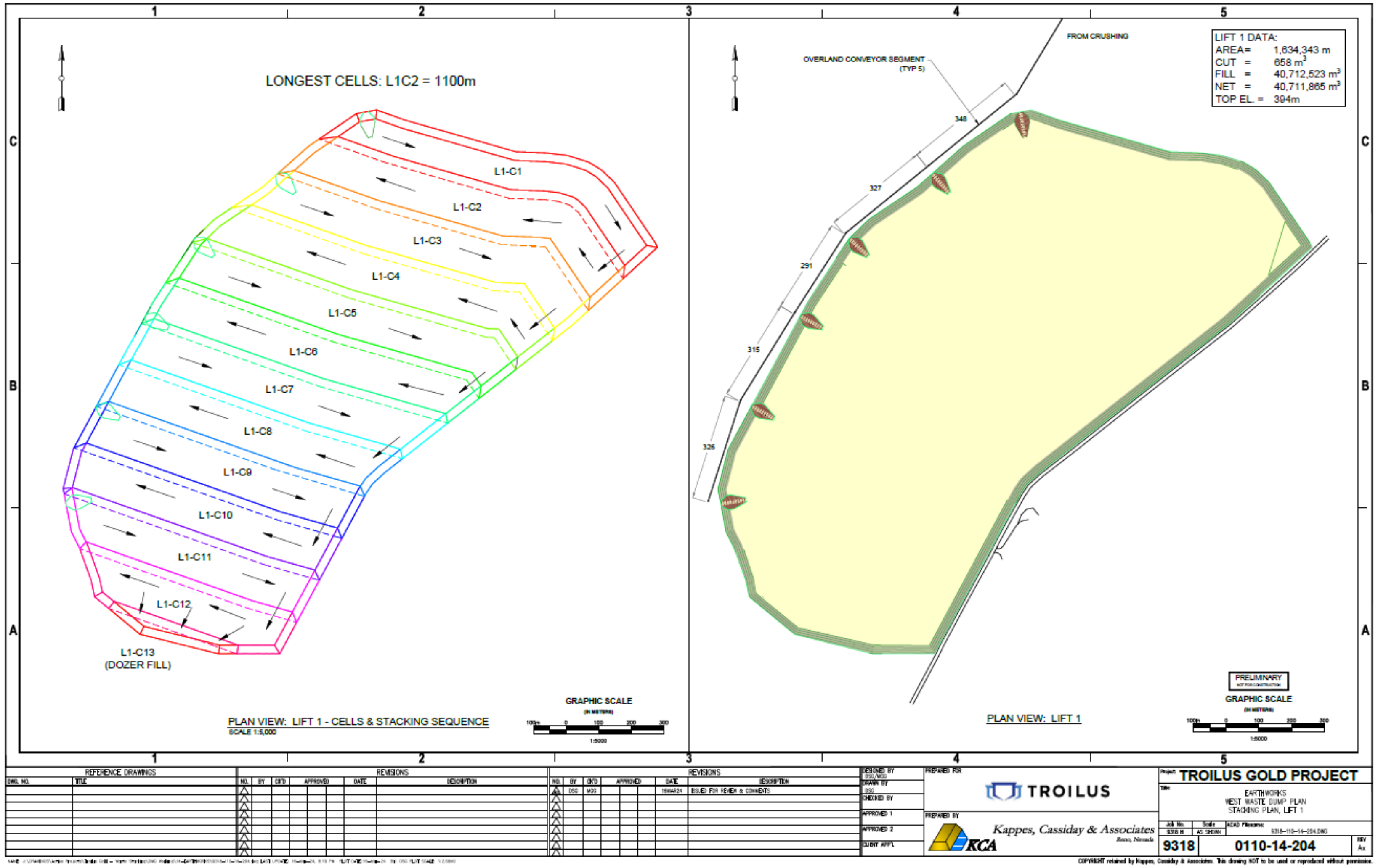
Table 24-12: West Waste Dump Lifts

	Waste Volume m^3	Waste Volume t	Waste Volume years
lift 1	40.7 M	85.1 M	2.1
lift 2	31.9 M	66.6 M	1.7
lift 3	28.3 M	59.1 M	1.5
lift 4	22.6 M	47.2 M	1.2
lift 5	17.4 M	36.3 M	0.9
lift 6	12.5 M	26.0 M	0.7
Total	153.3 M	320.3 M	8.0

Table 24-13: West Waste Dump Annual Stack Tonnes

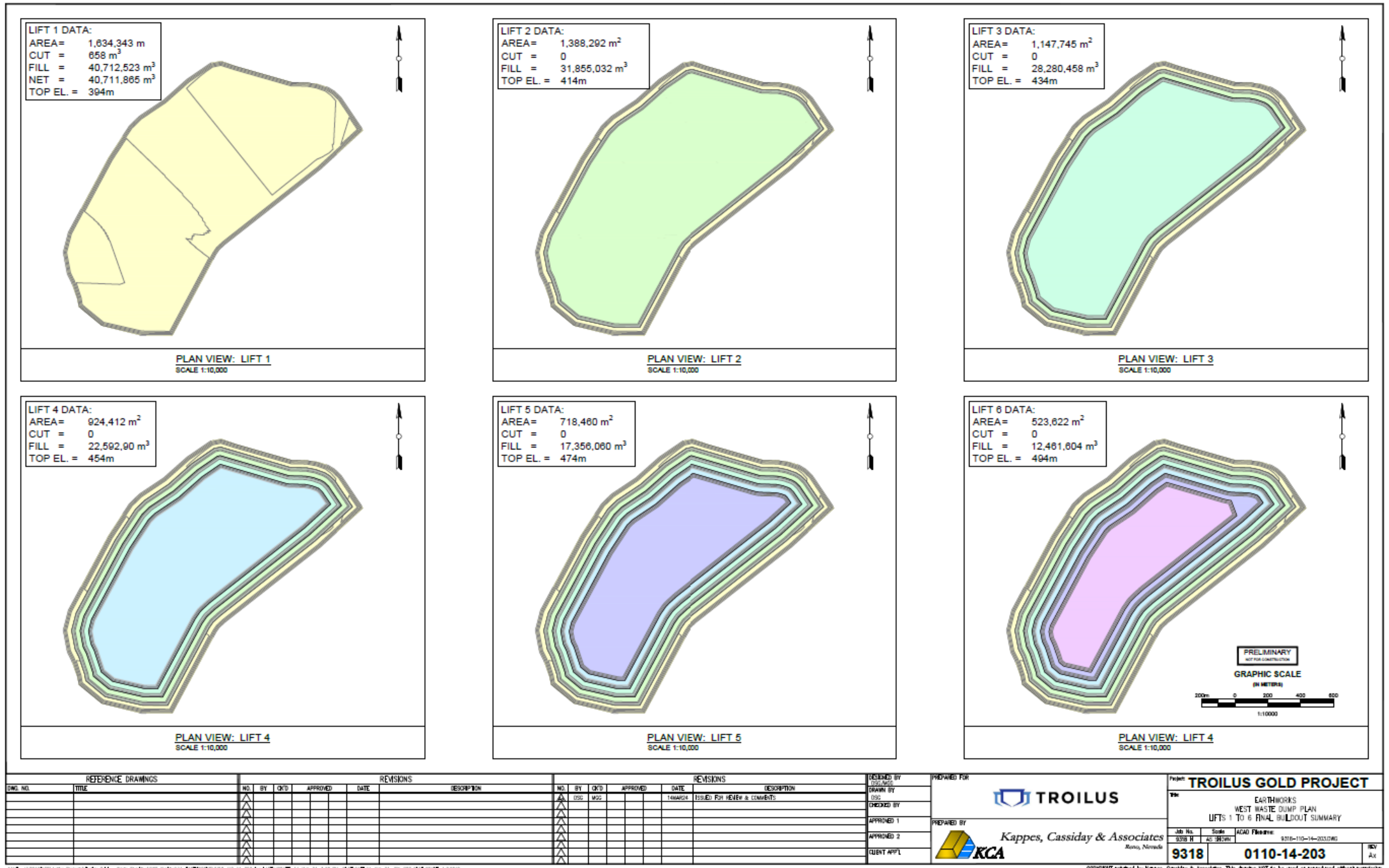
	West Waste Dump Stacked Tonnes	
	Annual	Cumulative
Year 2	40.0 M	40.0 M
Year 3	40.0 M	80.0 M
Year 4	40.0 M	120.0 M
Year 5	37.8 M	157.8 M
Year 6	27.8 M	185.5 M
Year 7	37.0 M	222.5 M
Year 8	40.0 M	262.5 M
Year 9	40.0 M	302.5 M
Year 10	17.8 M	320.3 M

Figure 24-4: West Waste Dump Stacking Plan



Source: KCA (2024)

Figure 24-5: West Waste Dump Stacking Plan by Lift



Source: KCA (2024)

87-WD Waste Dump

The North Waste Dump will be stacked in 20 m lifts. The east side of the dump will be truck stacked with 272.2 M tonnes (130.2 M m^3) of waste. The west side of the dump will be conveyor stacked with 408.7 M tonnes (195.6 M m^3) against the truck dumped waste.

The stacking conveyors will operate as described in Section 0.

The stacking will progress from the near center of the 87-WD and progress towards the northwest of the dump.

The waste stacked by year is shown in Table 24-14 below.

Table 24-14: North Waste Dump Annual Stacked Material

	Waste Tonnes			Waste Volume, m^3		
	Truck Dumped	Conveyed	Total	Truck Dumped	Conveyed	Total
Year 10	31.5 M	22.2 M	53.7 M	15.1 M	10.6 M	25.7 M
Year 11	12.9 M	40.0 M	52.9 M	6.2 M	19.1 M	25.3 M
Year 12	10.5 M	40.0 M	50.5 M	5.0 M	19.1 M	24.1 M
Year 13	19.0 M	40.0 M	59.0 M	9.1 M	19.1 M	28.2 M
Year 14	16.1 M	40.0 M	56.1 M	7.7 M	19.1 M	26.8 M
Year 15	10.9 M	40.0 M	50.9 M	5.2 M	19.1 M	24.4 M
Year 16	9.2 M	40.0 M	49.2 M	4.4 M	19.1 M	23.5 M
Year 17	26.6 M	10.0 M	36.6 M	12.7 M	4.8 M	17.5 M
Total	**136.5 M**	**272.2 M**	**408.7 M**	**65.3 M**	**130.2 M**	**195.6 M**

24.7 Evaluation of Underground Caving Potential

A high-level mine planning work evaluation was completed to evaluate the potential of sublevel caving or block caving to replace the Phase 3 open pit pushback of the 87 ore body considering only Measured and Indicated Mineral Resources. Work completed was limited in scope and should not be considered an exhaustive evaluation of potential opportunities.

While the results of this investigation should not be considered conclusive, they should generally be considered modestly optimistic as the optimizations utilized unconstrained footprints, and the results/inventory qualities would be expected to degrade moderately when practical mining boundaries are applied to the optimized areas.

The technical evaluations completed to assess the 87 ore body in this study involved using both Geovia Footprint Finder and Hexagon MinePlan software.

MinePlan was used to manipulate the resource block model and to export shape files to Footprint Finder. Footprint Finder was used to evaluate both an SLC mixing model for underground mining, and a block cave option. The Footprint Finder planning framework utilizes the calculated block values and subtracts an underground mining cost from them (assumed to be \$22.5/ for SLC and \$12.5/t for block caving).

The block cave optimization results do not suggest a viable case for extracting the 87 zone with a conventional block cave due to low footprint inventories and values.

The sublevel caving results are slightly more positive, though the initial runs showed a very marginal mining target at the metal prices considered. It is understood that the results were inferior to the value generated by open pit mining options, and as such were not recommended to consider further the replacement of Phase 3 with a sublevel cave mine.

One potential upside which could change this result in the future is considered successful infill drilling at depth. Currently there are pockets of inferred material which are breaking up the SLC mining shapes, but these could potentially be converted to improve continuity and value of the SLC inventory, which might help to justify further study into the potential of trading off the open pit with a larger SLC mine. Further refinement to consider a sub-level shrinkage with open pit waste backfill, similar as to how it is employed at Lac des Ilses Mine might allow for pit ramps to be preserved, reducing development CAPEX.

25 INTERPRETATION AND CONCLUSIONS

25.1 Geology

The Project is made up of four principal mineralized zones: Z87 Zone, J Zone, X22 Zone, and SW Zone. The Z87 Zone and J Zone were subject to open pit mining operations between 1996 to 2010. It has been established that there are still significant open pit and underground mineral resources in these, and adjacent zones. The X22 Zone has been recently discovered and developed in 2023 and is situated adjacent to the southwest of Z87 Zone. The SW Zone, situated approximately 2.5 km southwest of the Z87 Zone, has been the focus of several drill campaigns since 2019 and has been established as a significant deposit for the Project. The gold grades within the interpreted mineralized domains are continuous and may still be open along strike and at depth.

The mineralized zones on the Property occur around the margins of the Troilus Diorite and comprise the Z87 Zone, J Zone, and X22 Zone. The SW Zone lies along strike and southwest of the Z87 Zone. Other important mineralization discovered on the Property to date include: the northern continuity of the J Zone, in the Allongé Target and Carcajou Target; and the north-western continuity of the SW Zone, toward Z87 Zone, the Gap Zone; and to the southwest of the SW Zone, in the Beyan and Cressida Targets. Additionally, Troilus has also investigated several regional exploration targets on the Property that include: the Testard Target, the Freegold-Bullseye Target, and the Pallador Target.

The Project is primarily a gold-copper deposit, but contains minor amounts of Ag, Zn and Pb, as well as traces of Bi, Te, and Mo. The gold and copper mineralization at the Troilus deposit comprises two distinct styles, disseminated and vein hosted. Gold mineralization is spatially correlated with the presence of sulphides, even though the sulphide content does not directly correlate with gold and copper grade. The matrix of the diorite breccia, the diorite and the felsic dikes represent the main host rocks for the mineralized intervals.

Between 2018 and August 2023, Troilus completed several diamond drill core programs which support the mineral resources along strike and at depth at the Z87 Zone, J Zone: X22 Zone and SW Zone. AGP is satisfied the drill programs conducted by Troilus on the Project meet industry standards and norms and that sample handling, preparation and analyses are appropriate for this style of deposit.

The Mineral Resources for the Project, amenable to open pit extraction, at a 0.3 g/t AuEQ cut-off grade are: Indicated Resource of 506.2 Mt at 0.57 g/t Au, 0.07 %Cu, 1.09 g/t Ag and 0.68 g/t AuEQ; and an Inferred Resource of 76.5 Mt at 0.53 g/t Au, 0.06 %Cu, 1.12 g/t Ag and 0.65 g/t AuEQ.

The Mineral Resources for the Project, amenable to underground extraction, at a 0.9 g/t AuEQ cut-off grade are: an Indicated Resource of 2.1 Mt at 1.35 g/t Au, 0.09 %Cu, 1.90 g/t Ag and 1.51 g/t AuEQ; and an Inferred Resource of 4.0 Mt at 1.36 g/t Au, 0.08 %Cu, 8.21 g/t Ag and 1.58 g/t AuEQ. The effective date of the Project Mineral Resources is 2 October 2023.

The quantity and grade of Inferred Resources reported above are conceptual in nature and 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. For these reasons, an Inferred Mineral Resource has a lower level of confidence than an Indicated Mineral Resource and it is reasonably expected the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. AGP is unaware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the mineral resource estimate.

AGP concludes that further development of the mineralized zones is warranted and recommended.

25.2 Geotechnical

The pit slope geotechnical recommendations presented in this report are based on a review of the slope geometries and slope performance of the Inmet pit slopes and by targeted geotechnical and hydrogeological investigations, laboratory strength testing, wall mapping, oriented core logging, optical and acoustic downhole televiewer surveys.

The stability analyses indicate that the rock mass is favourable to the development of steep inter-ramp slopes in all areas of the pits except for open pit east walls where the bench face angle is constrained by planar failure involving moderately dipping foliation. Recommended open pit east wall inter-ramp angles range from 45 to 50 degrees, except at X22 pit, where the foliation is shallower, and the recommended inter-ramp angle is 38 degrees. For other wall orientation, the recommended inter-ramp angles range from 50 to 57 degrees.

Building on the precedents and experience gained by Inmet operation while mining the historical 87 and J4 pits, the pit slope designs were optimized assuming that ground support will be used to achieve safe bench geometries. The implementation of the slope designs will require a dedicated team, strong procedures and systems and capability to adapt. It will require a high level of skill applied to perimeter blasting, scaling, ground support, depressurization, and slope movement monitoring by the mine team.

Unloading cuts may be required for the west walls of J4 and 87 to manage potential increased risks with pit deepening. This can be reassessed and managed as mining progresses. The hazard is potential increased risk of bench and multi-bench failures associated with the toppling controlled deep-seated slope relaxation in response to pit deepening at depth. This deformation mechanism is expected to be primarily driven by the presence of unfavourable major structures and high-water pressure in the wall. Recommendations were formulated for maximum slope heights and slope angle combinations to be reviewed during detail design and based on slope performance.

25.3 Mining

The life of mine plan is based on Probable Mineral Reserves of 380 Mt with grades of 0.49 gpt gold, 0.058% copper, 1.00 gpt silver. Waste to be moved is 1,171 Mt for an overall strip ratio of 3.1:1 (waste:ore).

The Troilus Gold Copper Project is planned to be an open pit operation using conventional mining equipment. Pit designs were developed for the J, 87, X22 and SW pit areas. The J pit design consists of two phases of successive pushbacks around the entire pit perimeter. The 87-pit design includes an initial phase 0 at the south end to assist with site water management, followed by phases 1 to 3 in the main portion of the pit. The X22 pit design consists of two phases, with slightly higher grades in the phase 1 at the south. The SW pit design consists of two phases which can be scheduled as satellite phases from the northern pits. The pit optimization shells used to guide the ultimate pits were also used to outline areas of higher value for targeted early mining and phase development. All pits were developed using 10 metre bench heights.

The mining rate of up to 87 Mtpa was selected based on strategic planning scenarios which demonstrated that the targeted mill capacity 50 ktpd (18.25 Mtpa) would be achieved. The mine will be in production for 21 years. A peak stockpile capacity of 48 Mt was reached near the end of year 11. Stockpile material is reclaimed from stockpiles after completion of mining and continues until early into the 22nd year.

The mining fleet will be comprised of up to three 165 mm down the hole drills, eight 203 mm down the hole drills, four 34 m^3 front shovels, and two 23 m^3 front end loaders. The truck fleet will total 41 – 227 tonne trucks with the usual assortment of dozers, graders, and other support equipment.

In addition, a small fleet will be used to construct the diversion ditch and portions of the prestripping and tailings management facility preparation. The costs for this are included in Infrastructure capital.

25.4 Mineral Processing and Metallurgical Testing

Findings from metallurgical testwork completed on recent samples for J, SW, 87 and X22 zones are as follows:

- hardness testwork results classified Troilus ore to be competent with A x b value of 26.0 at the 15th percent and 29.8 at the 50th percentile
- bond abrasion index measuring from 0.2 to 0.4 indicates that the ore is moderately abrasive
- crushing work index has been derived from A x b data to be 22.5 kWh/t
- bond ball mill work index of 13.8 kWh/t at the 85th percentile and 12.1 kWh/t at the 50th percentile
- locked cycle PILOTWAL HPGR testwork resulted in an average m·dot value of 270 $t{\cdot}s/m^3{\cdot}h$ at a net pressing force of 3.33 N/mm^2
- average gravity gold recovery (future) is expected to be ~32%
- optimum flotation grind size was P_{80} 75µm for primary milling and P_{80} 20µm for regrind milling
- further treatment (leaching) of the flotation tails is not required or justifiable economically due to low flotation tails grades
- based on LOM head grade of 0.49 g Au/t, 1.00 g Ag/t and 0.06% Cu, the following recoveries are expected:
  - 91.6% Cu, 91.9% Au and 86.6% Ag for J-Zone
  - 89.9% Cu, 87.4% Au and 82.7% Ag for SW-Zone
  - 91.8% Cu, 94.7% Au and 97.6% Ag for Zone 87
  - 94.5% Cu, 93.1% Au and 89.9% Ag for X22 Zone
- flotation reagent consumptions for all zones combined are approximately 56 g/t KAX, 32 g/t SPRI 206, 29 g/t frother and between 100 to 400 g/t Na_2SO_3 depressant

25.5 Infrastructure and Site Layout

25.5.1 Site Infrastructure

The overall facilities and major infrastructure cover the mine site area, TMF, WMF, camp site, main access road, mine haul roads, site buildings, STP, WTP, explosives storage facility and site wide water management systems.

At start-up, the construction/operations camp will accommodate 530 personnel at the peak of construction. All camp and catering facilities on a lease and per diem basis for the first 5 years. The facilities will include all accommodations, catering, lounges, and a fitness centre.

The Project is currently serviced by an existing 161 kV power line that is owned and operated by Hydro Québec, with power capacity of 75 MVA. Plant full load Maximum Demand (MD) is 84.66 MVA. The line will be upgraded for the additional load. Standby generators will provide power to the process plant and ancillary electrical equipment in the event of a utility power failure.

25.5.2 Site Wide Water Management

Water management is a critical part of a mine project's construction, operation, and closure. A water management plan, including a feasibility engineering design and site-wide water balance model, was completed in view of restarting mining operations on the Site. The water management plan aimed to facilitate efficient mine operations and reduce effects on downstream receiving waterbodies.

The proposed water management plan includes water management structures to construct over the LoM:

- Four stream diversion channels with a total length of 15.6 km, including a 9.7 km long channel representing the main diversion structure; and
- Twenty-five collection ditches with a total length of 18.9 km, eight collection sumps, which require approximately 261,000 m^3 total active storage volume, and four sedimentation ponds, which require approximately 421,000 m^3 total active storage volume; the indicated volumes do not consider dead or freeboard storage. One of the sedimentation ponds will make use of the southern extremity of the 87 Pit and will have an effective capacity greatly exceeding the minimum required; the other three ponds will be built by excavation in the natural ground.
- All collection sumps and sedimentation ponds include pumping systems to allow water management on site, including transfer between sumps and ponds, discharge from the sedimentation ponds to the environment, or return of collected water to the ore processing plant:
  - the sump pumping systems include locations with submersibles pumps except for one which will be self priming
  - the sedimentation ponds pumping systems have vertical turbine or low-lift pumps installed on barges or shore-mounted pumping wells and pipelines; the water collecting in the sedimentation ponds is pumped towards the environment if the water quality meets the discharge guidelines. In case of pump shutdown due to high TSS, a predefined time delay precedes the automatic restart of the pump sequence, allowing particles time to settle

The design criteria included the applicable requirements from the provincial Directive 019 for the mining industry and from the Canadian Dam Association technical bulletins. Generally, the 1:100-year flood event, with consideration of climate change, was used for the design of the water management structures. As an exception, the Directive 019 “project flood” and the probable maximum flood (PMF) were used for the design of the tailings management facility.

Alignments and footprints for all water management structures were selected in consideration for the planned site development and for the terrain topography and known surficial soil conditions. As much as possible, the design avoided the need for above ground retention berms and selected profiles that limited the required excavation volumes. All pipelines are above ground with profiles that allow complete drainage when pumping stops.

The main diversion channel was designed to allow for fish passage between Lake Amont, upstream of the site, and Lake A, downstream of the site.

The construction of these structures is planned to be staggered over the first few pre-processing and operational years to limit initial capital costs. The use of available water management infrastructure and equipment were considered. The construction staging was based on the available mine development schedule. A conceptual (qualitative) water management plan was proposed for the closure period.

The contact water surplus will be treated to limit the concentration of suspended solids before discharge to the environment. The treatment will either be passive in the sedimentation ponds or active with a flocculation/sedimentation plant for the water in the TSF Pond. Based on ongoing geochemical investigations, Troilus indicated that suspended solids are the only contact water quality parameter, which is expected to require treatment.

The water balance model indicated that the contact water management system could provide the required process water supply even when accounting for seasonal effects and variable climate conditions. The water balance analysis considered a large variety of climate conditions, both historical and including climate change projections.

25.5.3 Tailings Management Facility (TMF)

To support the tailings generated from the life of mine (LoM) operations, the existing TMF will be raised from the current crest elevation of approximately 399 to 401 masl (dyke height of approximately 30 m), to an ultimate elevation of 435 masl (total dyke height of approximately 64 m). The TMF raise will provide an additional 169 million tonnes (Mt) of storage capacity. The intent of the FS TMF design is to raise the existing TMF while limiting the additional required site footprint.

The FS design concept of the Main Dyke raise, and new East Dyke is for downstream construction. The design assumes a raise of the existing dyke to a final elevation of 435 masl and construction of a new saddle dyke along the low point of the mountain slope on the eastern side of the TMF, also to 435 masl.

For the Main Dyke in particular, measures have been incorporated into the design to migrate from the current upstream structure to downstream construction, resulting in improved geotechnical stability. The downstream construction method provides additional storage capacity and a lower risk of failure than the upstream and centreline construction adopted for the existing TMF.

The TMF embankment will include a zoned run-of-mine waste rock embankment with upstream sand and gravel filters underlain by a layer of geotextile. There will be surplus waste rock produced throughout the LoM operations (i.e., more than what is required for embankment construction), meaning there will be no material balance issues for sourcing construction fill. Filters will be produced on site through crushing and likely screening of locally sourced materials.

Deposition modelling was completed in annual or more frequent increments. The rate of rise for the facility is approximately 4 m/yr throughout the life of the facility, based on an annual tonnage of 18.18 Mt.

The feasibility design was developed using extreme loading conditions which meet or exceed the Global Industry Standard for Tailings Management (GISTM) (GTR, 2020) and the criteria of Directive 019, which outlines applicable controls for the design and the construction and operation of TMFs in Québec. Canadian Dam Association (CDA) dam safety guidelines are used broadly and depend on a risk-based classification. Consequence classification is determined for a TMF based on incremental losses that a failure of the TMF may inflict on downstream or upstream areas. As the earthquake and flood criteria will be consistent with GISTM, which presumes extreme loading events (which would support the design of an extreme consequence facility as defined in CDA), the requirements of the CDA and consequence classification were not defined for this study.

The FS embankment designs meet the geotechnical stability requirements under static and extreme seismic loading conditions. Stress-deformation analysis and simplified dynamic displacement analyses were completed and results indicate that the existing embankment filters will experience minor deformation and should perform as per their design intent.

Drainage from the tailings beach and mountain slope will be collected by gravity to the TMF pond. The drainage includes both natural runoff and water exfiltrating from the deposited tailings. Pond water will be pumped for recirculation to the process plant and, if there is surplus, for discharge to the environment after treatment for total suspended solids removal via a treatment plant. The TMF is required to operate with a minimum 250 m long upstream tailings beach and a minimum 1.5 m pond freeboard, both of which are achievable based on geometric modelling carried out for the study.

The pond and the beach geometry will allow containing the regulatory Directive-019 environmental "project flood." The emergency spillway will be designed to manage the probable maximum flood (PMF).

The TSF will reach its full capacity during year 10 and tailings are planned to be disposed into mined-out pits until the end of Mine Life (year 22). In-pit tailings disposal will progress sequentially into the Southwest pit, J4 pit and then 87 pit. The mine plan was developed with this constraint of concept for migration to in-pit disposal in the later years of mine development.

25.5.4 Water Treatment

To support the site water management plan, the project includes the replacement of the existing and aging water treatment plant (WTP). The WTP is used to lower the total suspended solids (TSS) in the excess water at its Tailings Storage Facility (TSF) that needs to be discharged to the environment. The capacity of the existing WTP is 1,200 m^3/h, which was deemed to be sufficient, based on the site wide water balance to respond to the needs of the Project. The expected TSS concentration is expected to be similar to those of the 1996-1997 operations start-up which were in the 150 mg/l with peaks up to 300 mg/L. While it is expected that the mill grind would be finer, with the potential to settle more slowly than historical tailings, the tailings will be thickened prior to discharge to the TSF, this has the potential to decrease the TSS in the TSF.

The WTP location selected is the north-east of the SW Pit, close to the treated water discharge location. The selected location is in an area between two waste rock piles and a natural waterway. This location is unlikely to be developed or otherwise impacted by possible mine expansion. It also reduced the required pipeline length and a slight decline from the TSF to the WTP reduces both capital and operating expenses.

The selected technologies, to replace the existing WTP remains the ballasted-flocculation lamella clarifier core water treatment technology for the removal of the TSS.

25.6 Environment and Permitting

Troilus Gold is currently working through the Federal and Provincial Environmental Processes. The Impact Study should be filed according to both sets of Guidelines, following regulations and best practices. Consultations are ongoing and communication with stakeholders will remain open through al the process.

25.7 Capital and Operating Costs

Detailed capital and operating cost estimates developed for the DFS including consideration for all direct and indirect costs associated with the Project. All costs were estimated in United States denominated currency unless otherwise noted. The cost estimates are sufficient for an AACEI Class 3 level of study.

Initial capital costs are estimated to be $1,074.6 million with a further $276.6 million of sustaining capital. The total life of mine project capital is forecast at $1,351.2 million. The initial capital includes $213 million in capitalized prestripping. The mining fleet is assumed to be financed and owner operated.

The estimate includes initial capital requirements of the mine, site, and tailings facilities. The Project benefits from having an existing permitted tailings facility currently on site. Sustaining capital needs for the open pits, process plant, infrastructure, and reclamation and closure costs are also included over the life of mine

All equipment and material are assumed to be new. Labour costs based on statutory laws governing benefits in effect at the time of the estimate have been included.

The project operating costs have been estimated at $17.24/t mill feed over the 18 year mine life. This includes all mining, processing, G&A, and concentrate trucking, port, and shipping costs. The costs were determined with a diesel price of $1.07/l and an electrical price of $0.26/MWhr.

25.8 Economic Analysis

The economic analysis, including taxation, show the Troilus Gold Copper Project has positive economics and technical merit.

A pre-tax and post-tax cash flow model in Excel was developed for the evaluation of the Troilus gold Copper Project FS. The following key parameters were used in the construction of the cash flow model and the economic results:

- Metal prices of:
  - Gold = $1,975 US/oz
  - Copper = $4.05 US/lb
  - Silver = $23 US/oz
- 100% equity financing with no debt component
- revenues and costs reported in constant H2 2023-dollar terms without escalation

The results indicate a post-tax NPV5% of $885 M and 14.0% IRR for the 21.1 year mine life. AISC for the project is $1,109/oz.

Total taxes paid life of mine is $1,342.0 million and is 37.8% of the pretax cashflow.

26 RECOMMENDATIONS

The QPs recommend that Troilus proceed with advancing the Project to Basic Engineering as part of the Project development plan. Recommendations and associated budgets are provided by the QPs to carry this work forward.

Estimated costs by area are provided in Table 26-1.

Table 26-1: Recommended Budget for Basic Engineering

Area of Study	Approximate Cost (C$)
Geology	$2,200,000
Geotechnical	$357,000
Mining	$150,000
Mineral Processing and Metallurgy	$25,000
Infrastructure (long lead items, engineering)	$63,700,000
Environmental	$2,750,000
TOTAL	**$69,182,000**

26.1 Geology

It is recommended that delineation drilling continue on all four mineralized zone of the Project to define the limits of each zone along strike. Approximately 2,000 m of drilling is proposed for the Z87 Zone and X22 Zone and the area between the two zones; and approximately 2,000 m of drilling for the J Zone. It is also recommended for 2,000 m of infill drilling be carried out at the SW Zone to target areas of Inferred resources in order to upgrade this material.

It is also recommended that the twinning of historic, pre-2018, drill holes, be targeted with more current drill information. Specifically, targeting drill holes with unsampled intervals at shallow depth and, where possible, to replace drill holes with unanalyzed silver assays. Approximately 3,000 m of drilling is proposed for this exercise.

It is recommended that bulk density and assay analysis for silver be completed for the initial drilling at Z87 Zone (approximately 4,000 samples). The early 2018 drilling did not include these analyses at the time and will be a necessary component to support silver mineralization at depth.

The following is the estimated budget for the proposed drilling programs for the continued development of the mineral resources. The estimated budget for these proposed exploration programs would be approximately C$ 2.2 million. Table 26-2 presents an estimated budget of the proposed exploration and development work.

Table 26-2: Estimated Budget – Geology

Description	Unit Cost (C$)	Estimated Cost ($)
Z87 Zone, J Zone, X22 Zone, SW Zone		
Diamond Drilling (6,000 m), delineation and infill	$200/m	$1,200,000
Z87 Zone, J Zone,		
Twinning of pre-2018 drill holes (~3,000 m)	$200/m	$600,000
Re-analysis (Z87 Ag analysis, bulk density); ~ 4,000 samples	$50/sample	$200,000
	Subtotal	**$2,000,000**
	Contingency	**$200,000**
	TOTAL	**$2,200,000**

26.2 Geotechnical

It is recommended to continue collecting oriented structural data in exploration holes. Foliation data near the east walls will be key in understanding local folding to plan for local adjustments to the pit design. A focus should be put on the northeast and southern portion of the J4 east wall, where major structures intersect the wall and shallow dipping foliation was measured in exploration holes and where the sharp variation in foliation orientation observed between exploration holes located close to each other requires investigation.

The following targeted geotechnical investigation and design update activities are recommended for the next stage of study:

- plan two 300 m deep geotechnical holes in areas where uncertainty in exploration data orientation could have impacts on the slope design, such as the southeastern and northeastern portion of J4 pit
  - allow 150 000$
- plan for two 200 m long geotechnical boreholes to improve characterization at the SW pit and to distinguish weaker conditions observed in the felsic and breccia units from potentially stronger and better quality intermediate volcanics (west wall) and mafic volcanics (east wall)
  - allow 100 000$
- complete 10 additional direct shear tests and further refine the shear strength of the discontinuity sets at the SW pit and determine whether or not weaker strengths should be used
  - allow 7 000$
- review oriented structural data collected after 2023 and its implication on slope designs, particularly for the east walls of the pits
  - allow 100 000$ for a review of the data, structural model update, design update, and potential added discretization of the design domains that the new data would allow

Total cost estimate of $357,000 for additional geotechnical work is anticipated during detailed design. During the early years of the operation in each pit, install piezometers on the west walls where bench inter-ramp and overall stability may be sensitive to groundwater pressures, anisotropy block passive drainage, and overall slopes may require unloading cuts as a result. Piezometers should be installed behind the ultimate east wall location of each pit. For budgetary purposes, plan for twelve grouted VWP hole 200 to 300 m deep with three vibrating wire piezometers each at X22, 87 and J4 pits and three at SW pit.

Additional geotechnical recommendations for future geotechnical work, including ground support, slope monitoring, dewatering and operational considerations are summarized in Chapter 16 and detailed in the technical report for geotechnical pit slope designs of 87, J4 and SW pits (WSP 2024a) and X 22 pit (WSP 2024b).

26.3 Mining

In the additional study is required in the following areas above the normal mine planning activities for an FS:

- Blast Optimization - $30,000
  - fine tune explosives with detailed rock information to reduce consumable costs
- Equipment Selection - $100,000
  - review equipment selection and fine tune drill size, shovel buckets and truck boxes to handle the abrasive materials and maximize carrying capacity/reduce unit costs
- Shovel Bucket Grade Control - $20,000
  - examination of shovel bucket-based technology to accurately track grade from face to mill

Total cost estimate of $150,000 for additional mining work is anticipated.

26.4 Mineral Processing and Metallurgical Testing

Additional testwork is recommended to enhance confidence in the regrind circuit design. This should include:

- settling and rheology testing on flotation concentrates
- regrind milling testwork to obtain a signature plot for the flotation concentrates

The cost associated with these additional testwork is estimated to be $25,000.

Other recommendation include consultation with a concentrate marketing specialist to advise on current penalty and payment terms for minor elements.

26.5 Infrastructure

26.5.1 Site Infrastructure

In the Basic Engineering Stage, it is recommended that the process, mechanical and electrical engineering be advanced to allow the procurement of long lead mechanical and electrical equipment packages and to obtain the vendor data to further the design of the concrete and steelwork in the critical areas. To obtain the vendor data, orders will have to be placed and it is expected that initial payments and the 1st installment will be due. The costs for these payments are included in the capital cost estimate.

- down payment / Initial Payment to procure long lead packages – USD 21.1 million (CAD 28.5 million)
- 1st Installment to procure long lead packages – USD 23.1 million (CAD 31.2 million)
- commencement of Basic Engineering to support long lead item procurement – USD 3.0 million (CAD 4.0 million)

26.5.2 Site Wide Water Management

The feasibility study engineering analyses for the water management structures should be updated and refined during the next engineering phase. Further integration between water management and mine, waste, and tailings management plans for the construction, operational, and closure periods are recommended. Recommended optimization activities for the next engineering phase include:

- Review of alignments, profiles, and footprints for channels, ditches, sumps, and ponds, to increase hydraulic performance and limit construction costs.
- For the pumping systems, exploration of opportunities to optimize pumping capabilities is encouraged. This could involve minimizing the number of pumped sumps required by implementing a partly buried pipeline, allowing gravity flow to other collection ponds. A trade-off would need to be undertaken to consider all aspects of such modifications, like excavation costs and accessibility of these systems over the life of mine. System redundancy should be further evaluated in the objective of reducing costs. A trade-off could be completed to evaluate the system's acceptable redundancy and costs savings.

In preparation for the next engineering phase for the water management structures, WSP recommends completing the following studies:

- Extend and improve geotechnical characterization along the water management structures footprint.
- Extend and improve the available geochemical characterization of the various water streams to be managed by the future Troilus mine operations. These streams include waste rock and tailings runoff and seepage, groundwater seepage to pits and ponds, and natural runoff. The improved characterization is required to verify major water management assumptions.
- Further refine hydrology and aquatic baseline studies to confirm and optimize the fish passage and fish habitat approach for the Main Diversion Channel.
- Develop progressive reclamation and mine closure plans.
- Depending on the schedule of the next engineering phase, an update of the climate baseline analysis to consider the most recent climate change projections may be recommended.

26.5.3 Tailings Management Facility

Recommendations related to the next stages of Tailings Management Facility are as follow:

- Identify alternative tailings option storing capacity to cover a shortage of approximately 15 months (22 Mt) occurring around year 17 based tailings production, mine plan and the planned tailings storage capacity.
- Conduct a dam safety review and dam breach analysis for detailed engineering design and revisit the dam consequence classification.
- Review potential borrow sources for the Tailings Storage Facility filter zone material and confirm quantity available.
- Collect more field data to better understand the thickness and properties of the foundation materials for the east dyke.
- Perform sensitivity analysis on the compressibility of the waste rock and the existing tailings among other parameters to inform the static deformation assessment.
- Develop a site-specific seismic hazard assessment to establish the earthquake peak ground acceleration and scaled ground motion time histories and to revisit the liquefaction assessment, dynamic stress-deformation analyses with these updated parameters.
- Confirm the geochemistry of tailings and waste rock, design changes could be required if the tailings or waste rock are reclassified as high-risk according to Directive 019 classification.

26.5.4 Water Treatment

It is recommended to confirm the water treatment design assumptions during the next stages of design to confirm water treatment requirements remain those of the existing water treatment plant. For example, the opportunity to decrease the plant's treatment capacity should be investigated. Also, the inflow suspended solids concentrations should be confirmed. Finally, the most recent available geochemical studies should be used to confirm that the suspended solids are the only water quality parameter that requires treatment.

26.6 Environmental

It is recommended to continue to gather data on physical and biological components that are currently being monitored to look out for trends and changes related to climate evolution. It is recommended to keep communication open with the stakeholders. Close attention should be given to the evolving Federal Assessment Process following the Supreme Judgment in October 2023 for filing the impact study to both level of Governments. Preparation of Construction and Operation permits should begin after filing the impact study.

Permits will require an additional USD $0.2 million (CAD $0.25 million) of expenditure.

The EIA process is expected to continue post-FS and with a cost to complete expected at USD 1.85 million (CAD $2.5 million).

27 REFERENCES

Asneimer, J., FLSmidth Mining Technologies GmbH (Research Center, Germany), 2024, High-Pressure Grinding Abrasions Testwork for Project.

Augustin, Jérôme., Rapport des travaux statutaires – Automne 2018. Propreté Trolus. Baie-James, Québec, Canada, S.N.R.C. 32J/15, 32J/16, 32O/01, 32O/02. Coordonnées du centre (UTM N83 Zone 18): 535 085 E / 5 649 950 N. Report. Prepared by Laurentia Exploration Inc. (Compagnie de services géologiques). 205 pages.

Bawden, October 2005. "Inmet Mining Corp. 12/7/2005 Troilus Mine Underground Geomechanical Feasibility Study".

Boily, B., 1995, The Lac Troilus Cu-Au Deposit, in Metallogenic evolution and geology of the Chibougamau area – from porphyry Cu-Au-Mo to mesothermal lode gold deposit, Geological Survey of Canada, Open File 3143, Precambrian '95 field trip guidebook, pp 123-130.

Boily, B., 1998, Le Gisement Cu-Au Troilus, in Géologie et métallogénie du district minier de Chapais-Chibougamau, nouvelle vision du potentiel de découverte, Ministère des Ressources naturelles, Québec, DV 98-03, pp 137-146

Boily, B., 1998, Porphyry-type mineralization in the Frotet-Evans Greenstone Belt – The Troilus Cu-Au Deposit, in Field trip guidebook, Post-meeting field trip B3: Metallogeny of the Chibougamau District: Geological Evolution and Development of Distinct Mineralized Systems through Time, for "Québec 1998", GAC/MAC/APGGQ, 1998

Boily, M., Dion, C., 2002, Geochemistry of boninite-type volcanic rocks in the Frotet-Evans greenstone belt, Opatica Subprovince, Québec: implications for the evolution of Archaean greenstone belts: Precambrian Research, v. 115, p. 349–371.

Brassard, B. 2018, Vieille Mine – Nouvelle Perspective: Old Mine – New Perspective. Conférence at the Xplor 2018 Convention, Montréal, October 17th.

Brassard, B., Hylands, B., 2019, The Troilus deposit: a new structural model and a new perspective. Conference at the Gac-Mac-IAH convention: Où les géosciences converges – Where Geosciences Converge. Session SY-RE01 - Gold: Recent advances, and 20 years of research through GSC's Targeted Geoscience Initiative Program under Dr. Benoît Dubé's leadership. May 13th, 2019, Québec City.

Brisson, H., Gosselin, C., Fallara, F., Gaulin, R., Dion, D.J., 1998a. Géologie de la région du lac Théodat (SNRC 32K/16). Ministère des Ressources Naturelles du Québec. RG 98-07, 24 pp.

Brisson, H., Gosselin, C., Fallara, F., Gaulin, R, Dion, D.J., 1998b. Géologie de la région du lac Evans (SNRC 32K/ 15). Ministère des Richesses Naturelles du Québec. RG 98-06, 23 pp.

Brisson, H., Gosselin, C., Fallara, F., Gaulin, R, Dion, D.J., 1998c. Géologie de la région du lac Rocher (SNRC 32K/ 09). Ministère des Richesses Naturelles du Québec. RG 98-05, 22 pp.

Bruin, C. and Pojhan, A., Base Metallurgical Laboratories Ltd., 2022, Supplemental Metallurgical Testing of the Troilus Gold Project – BL0803.

Carles, P., 2000, Constraint on the Genesis of the Archaean Troilus Gold-Copper Deposit, Québec, master's degree thesis, Department of Earth and Planetary Sciences, McGill University, Montreal, December 2000

Carson, D.J.T., 1990, Mineralogy and predictive metallurgy of the Frotet Lake deposit, Noranda Exploration Company Limited, report submitted to Minnova Inc., Dated May 4, 1990.

Centre de Recherche Minérale du Québec, 1991, Étude de la dispersion de la minéralisation de la propriété Frotet-Troilus, par Jean-François Wilhelmy, Centre de Recherche Minérale, projet 90LP107, rapport final, submitted to Minnova Inc., dated April 3, 1991.

Centre de Recherches Minérales (CRM), St Foy, Québec (1991) – Preliminary bench scale metallurgical testwork. (Details of this testwork were not available for review at the time of preparing this summary).

Chryssoulis, S.L., 1990, Technical Report on the 'Invisible' gold content of sulphide minerals from the Frotet Lake prospect, Québec; Process Mineralogy, University of Western Ontario, Report submitted to D. Carson, Noranda Exploration Company Limited, 1990.

COREM, St Foy, Québec (2019) – Bottle roll testwork on 6 composites at various crush and grind sizes.

COREM, St Foy, Québec (2020) – JK Dropweight testing of four domain composites (Domain 41, Domain 42, Domain 43, and Domain 54)

Davis, W.J., Machado, N., Gariépy, C., Sawyer, E.W., 1995, U–Pb geochronology of the Opatica tonalite-gneiss belt and its relationship to the Abitibi greenstone belt, Superior Province, Québec, Can. J. Earth Sci., v.32, p.113–127.

Diniz, T.B., 2019, An Overview of the Troilus Au-Cu Deposit, Frotet-Evans Greenstone Belt, Opatica Subprovince, Québec: Unpublished M.Sc. thesis, Kingston, Ontario, Canada, Queen's University, 85 p.

Dion, C., Pilote, P., David, J., and Cater, D. 1998. Géologie et âge de la minéralisation Au-Cu a la mine Troilus, ceinture de Frotet Evans, Québec [abs]. Geological Association of Canada Annual Meeting, Abstracts Volume 23, p. 46.

ECCC. 2020. Wind Speed Statistic Analysis – Wind Roses and Extreme Wind Speed Frequency. Climate Québec. Electronic communication, December 2020, John Richard on behalf of ECCC. (ec.climatQuébec-climateQuébec.ec@canada.ca).

ECCC. 2023. Climate Data for a Resilient Canada. Downloaded from climatedata.ca in August 2023.

Fan, M., Eriez Flotation, 2022, Troilus Gold Corporation – Copper/Gold Flotation Investigation Report (for J, SW & 87 3,000 kg Samples).

Fan, M., Eriez Flotation, 2023, Troilus Gold Corporation – Copper/Gold Flotation Investigation - J-Zone, SW-1, SW-2 and 87 Zone Report (for 800 kg Samples).

Fan, M., Eriez Flotation, 2023, Troilus Gold Corporation – Copper/Gold Flotation Investigation – X22-1 and X22-2 Report (for 150 kg Samples).

Fan, M., Eriez Flotation, 2024, Troilus Gold Corporation – X22-1 and X22-2 Zone Flotation Investigation Report.

Fraser, R.J., 1993, The Lac Troilus gold-copper deposit, Northwestern Québec: A possible Archean porphyry system, Economic Geology, 88: pp 1685-1699.

Genivar, 2009. "Plan de fermeture et de restauration du site Troilus", 482 pages.

Global Tailings Review (GTR). 2020. Global Industry Standard for Tailings Management.

Goodman, S., Williams-Jones, A.E., and Carles, P., 2005, Structural controls on the Archean Troilus gold-copper deposit, Québec, Canada, Economic Geology, 100: pp. 577-582.

Gosselin, C., 1990, Géologie de la partie NE de la région de Frotet-Troilus, Ministère de l'Énergie et des Ressources du Québec, DV 90-40, pp 65-67.

Gosselin, C., 1996, Synthèse géologique de la région de Frotet-Troilus, Ministère des Ressources naturelles, Québec, ET 96-02.

Gosselin, C., 1998, Synthèse géologique de la région de Frotet-Troilus, Géologie et métallogénie du district minier de Chapais-Chibougamau, nouvelle vision du potentiel de découverte, Ministère des Ressources naturelles, Québec, DV 98-03, pp 63-70.

Hamand, E. B., Hazen Research, Inc., Golden, Colorado, 2024, Comminution Testing Results for Troilus Sample (Project No. 13134) - Abrasion, Bond Ball Mill Work Index, and SMC Testwork.

Hazen Research, Golden, Colorado (2020) – Abrasion Index and Bond Ball Work Index testing on three composites (87UG, 87S and SW).

Itasca Consulting Group (Andrieux), August 2005. "Report on the Site Visit of 2-3 August 2005 at Troilus Mine"

Itasca Consulting Group (Andrieux), January 2006. "Report to Inmet Mining Corporation, Ltd. Troilus Division Geomechanics Assessment With 3DEC Numerical Analyses of Underground Stoping Options at the Troilus Division"

Inmet Mining, March 2006. "Troilus Underground - Technical Report - March 2006"

J-Consultores Ltda., 2024, Criteria for 50 ktpd Design Capacity for Troilus Project (Third Party Review).

Kappes Cassidy & Associates, Reno, Nevada, 2020, J4 Zone and J5 Zone Composite Samples Bottle Roll Leach Testing (Report ID: KCA0200020_TROI01_03).

Kappes Cassidy & Associates, Reno, Nevada, 2020, J4 Zone and J5 Zone Composite Samples Conventional and HPGR Crushed Bottle Roll Testing (Report ID: KCA0200021_TROI02_02).

Kappes Cassidy & Associates, Reno, Nevada, 2020, J4 2020 and J5 2020 Composite Samples Conventional and HPGR Crushed Leach Testing (Report ID: KCA0200022_TROI03_02).

Kappes Cassidy & Associates, Reno, Nevada, 2020, 87OP, 87UG, 87S and SW Samples Flotation and Bottle Roll Leach Testing (Report ID: KCA0200023_TROI04_03)

Kappes Cassidy & Associates, Reno, Nevada, 2024, Composite 01 and Composite 02 HPGR Single Pass and Locked Cycle Testwork (Report ID: KCA0230028_TRO08_01)

Lakefield Research, Lakefield, Ontario (1993) – Bench scale and pilot plant testwork focused on producing a copper-gold flotation concentrate.

Lakefield Research, Lakefield, Ontario (2003) – Bench scale gravity concentration and flotation testwork on 3 composites from the J4 Zone.

Leroux, J., Email: Acid Base Accounting Testwork, November 2023, Attachment – Compilation_Caract_Tails_Geochime_Troilus.xlsx

Lowel, J.D., and Guilbert, J.M., 1970, Lateral and vertical alteration – mineralization zoning in porphyry ore deposits, Economic Geology, 65: pp 373-408.

Lycopodium, 2023. Troilus Gold Project DFS Site Visit Report. 14 Pages. 29 Sept 2023.

McMillan, W.J., and Panteleyev, 1987, A. Porphyry copper deposits, in Ore deposit models, Geoscience Canada Reprint Series, Edited by R.G. Roberts and P.A. Sheahan, vol. 3, pp45-58.

Morin, R., 1998a. Géologie de la région du lac Salamandre. Ministère des Richesses Naturelles du Québec. RG 97-14, 16 pp.

Morin, R., 1998b. Géologie de la région du lac Ou âgama. Ministère des Richesses Naturelles du Québec. RG 97-15, 18 pp.

Morin, R., 1998c. Géologie de la région de Chutes aux Iroquois. Ministère des Richesses naturelles du Québec. RG 97-16, 15 pp.

Morin, R., et al., 1999, Potentiel minéral du district minier de Chibougamau, Ministère des Ressources naturelles, Québec, PRO 99-02. Pilote P., et al., 1997, Géochronologie des minéralisations d'affiliation magmatique de l'Abitibi, secteur Chibougamau et de Troilus-Frotêt: implications géotechniques, Ministère des Ressources naturelles, Québec, DV 95-04, p 14.

MELCCFP (Ministère de l'Environnement, de la Lutte contre les changements climatiques, de la Faune et des Parcs). 2012. Directive 019 pour l'Industrie Minière.

Ouranos. 2024. Regional Climate Maps. Consortium on Regional Climatology and Adaptation to Climate Change. Consulted in December 2020 at https://www.ouranos.ca/climate-portraits/#/.

Pitard, F.F., 1999, Study of the heterogeneity of gold in the Troilus mine J4 ore, Francis Pitard Sampling Consultants, report submitted to Inmet Mining Corp., Dated February 5, 1999.

Piteau Associates, September 2006. "Troilus Project September 2006 – Rock Mass Parameters for Numerical Modeling of Proposed Underground Mine", by P.M. Hawley, File No. 1495.

Piteau Associates, October 2006. "Troilus Pit October 2006 Geotechnical Inspection – Draft", by P.M. Hawley, File No. 1495.

RPA Consulting. June 2018. "Draft Memo Re: Mining Methods Review Date June 7, 2018 (13:30)"

Riverin, G., 1997, Étude de la granulométrie de l'Or dans les échantillons de Troilus, Internal Memorandum, Inmet Mining Corporation, Dated March 20, 1997.

SGS Lakefield Research, Lakefield, Ontario, 2003 – An Investigation into The Metallurgical Response of J4 Ore (Report No. LR10484-001).

Simard, A., 1987, Stratigraphie et volcanisme dans la partie orientale de la bande volcano-sédimentaire archéenne Frotet-Evans, Ministère de l'Énergie et des Ressources du Québec, MB 87-17.

SRK Consulting Inc. November 2018. "Structural Investigation Report of the Troilus Project, Québec"

SRK, 2018, Structural Investigation of the Troilus Project, Québec. Internal Report Prepared for Troilus Gold Corporation by Caté, A., November 28. 43 pages.

Stantec, 2023. Rapid Voltage Change Study for the proposed expansion of Chibougamau Mine. 45 Pages. 30 June 2023

Stepperud, E. C., Hazen Research, Inc., Golden, Colorado, 2020, Comminution Testing (Project No. 12751) - Abrasion Index and Bond Ball Work Index testing on Three Composites (87UG, 87S and SW).

Sutherland Brown, A., 1976, Morphology and classification, Canadian Institute of Mining and Metallurgy, Special Volume 15, pp 44-51.

Ticzon, M., FLSmidth Minerals Testing and Research Center, 2023, Gravity Recoverable Gold Test Report (Project Reference: P-23014).

Tremblay-Bouliane, K., COREM, 2019, Evaluation of Gold Recovery from the Cyanidation of Assay Core Rejects Associated with the Troilus Gold Project (Report No. T2530).

Tremblay-Bouliane, K., COREM, 2021, Physical and Chemical Assessments of the Troilus Deposit (Report No. T2779).

Troilus Gold Corp. November 2019. "Troilus Évaluation environnementale - Dénoyage des fosses J-4 et 87"

Troilus Gold Inc. (Troilus). 2022. Draft version of a Preliminary Report Summarising Environmental Studies, Permitting and Social or Community Impact for the Troilus Gold Project. Electronic Communication from Jacqueline Leroux (Troilus) on July 20, 2022.

Westwood, J., FLSmidth Minerals Testing and Research Center, 2024, Gravity Recoverable Gold Test Report (Project Reference: P-23059).

Wilson, M., FLSmidth Minerals Testing and Research Center, 2021, Gravity Recoverable Gold Test Report (Project Reference: P-21102).

Wilson, M., FLSmidth Minerals Testing and Research Center, 2021, Gravity Recoverable Gold Test Report (Project Reference: P-21076).

WSP. 2024a. Feasibility Level Open Pit Slope Design Recommendations Pits J4, 87 and South-West - Troilus Gold Mine Doc ref. 22538260-007-R-Rev0. Dated June 21, 2024.

WSP, 2024b. Pit X22 Feasibility Level Design of Open Pit Slope Angles, Doc ref. 22538260-010-R-Rev0 6000. Dated June 20, 2024.

WSP, 2024c. Troilus Project Operational Site-Wide Water Management Plan – Feasibility Study, Doc ref. 059-2257554004-Rev0. Dated June 2024.

WSP. 2024d. Modélisation hydrogéologique des fosses projetées du projet Troilus. Doc ref. 22538260-011-R-Rev0. May 9, 2024.

WSP, 2024e. Troilus Project Feasibility Study – Design of water pumping system included in the site-wide water management plan, Doc ref. 056-2257554004-Rev0. Dated June 2024.

WSP, 2024f. Troilus Gold water treatment plant relocation and replacement Feasibility Study, Doc ref. 062-2257554003-TM-Rev1. Dated May 27, 2024.

WSP, 2024g. Tailings Management Feasibility Study – Troilus Feasibility Study, Doc ref. 055-2257554003-Rev0. Dated June 2024.

Weier, M., JKTech, 2020, JKDW Test Report (Job No. 20114/P3) - JK Dropweight Testing of Four Domain Composites (Domain 41, Domain 42, Domain 43, and Domain 54)

Technical Reports

AGP, 2023. Technical Report and Mineral Resource Estimate on the Troilus Gold-Copper Project Québec, Canada. 343 pages. 2 October 2023.

AGP, 2020a. Technical Report and Mineral Resource Estimate on the Troilus Gold-Copper Project Québec, Canada. 212 pages. 27 August 2020.

AGP, 2020b. Preliminary Economic Assessment of the Troilus Gold Project. Québec, Canada. 483 pages. 14 October 2020

RPA, 2014. Technical Report on the Troilus Gold-Copper Mine Mineral Resource Estimate, Québec, Canada. 30 June 2014.

RPA, 2016. Technical Report on the Troilus Gold-Copper Mine Mineral Resource Estimate, Québec, Canada. 30 June 2016.

RPA, 2017. Technical Report on the Troilus Gold-Copper Mine Mineral Resource Estimate, Québec, Canada. 20 November 2017.

RPA, 2019a. Technical Report on the Troilus Gold-Copper Mine Mineral Resource Estimate, Québec, Canada. 1 January 2019.

RPA, 2019b. Technical Report on the Troilus Gold-Copper Mine Mineral Resource Estimate, Québec, Canada. 20 December 2019

Websites

Gestim Plus. Online Register of mineral titles website. Ministère de l'Énergie et des Ressources Naturelles.

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

(most recently viewed 2 October 2023)

Troilus Gold Website, Press Releases:

https://www.troilusgold.com/news-and-media/news-releases/

(most recently viewed 2 October 2023)

Press Release 24 May 2018 Z87 Zone, drill intercepts
Press Release 9 July 2018 Z87 Zone, drill intercepts

Press Release 12 September 2018 Z87 Zone, drill intercepts
Press Release 31 October 2018 Z87 Zone, drill intercepts
Press Release 19 August 2019 Z87 Zone, drill intercepts
Press Release 17 August 2022 Z87 Zone, drill intercepts
Press Release 26 March 2019 J Zone, drill intercepts
Press Release 12 May 2021 J Zone, drill intercepts
Press Release 8 June 2021 J Zone, drill intercepts
Press Release 7 July 2021 J Zone, drill intercepts
Press Release 21 September 2021 J Zone, drill intercepts
Press Release 7 July 2021 X22 Zone, drill intercepts
Press Release 31 March 2023 X22 Zone, drill intercepts
Press Release 12 January 2021 SW Zone, drill intercepts
Press Release 9 February 2021 SW Zone, drill intercepts
Press Release 24 February 2021 SW Zone, drill intercepts
Press Release 16 March 2021 SW Zone, drill intercepts
Press Release 17 August 2021 SW Zone, drill intercepts
Press Release 20 January 2022 SW Zone, drill intercepts
Press Release 21 April 2022 SW Zone, drill intercepts
Press Release 4 May 2022 SW Zone, drill intercepts

Québec Mining Act. (English):

http://legisQuébec.gouv.qc.ca/en/ShowDoc/cs/M-13.1

Worldclimate.com website: Mistissini, Québec:

http://www.worldclimate.com/cgi-in/grid.pl?gr=N50W073

(most recently viewed 2 June 2020)

Yr.no website; Norwegian Meteorological Institute and the Norwegian Broadcasting Corporation: Mistissini, Mont Wapaskouch, Pic Longview; Québec:

https://www.yr.no/place/Canada/Québec/Mistissini~6075492

https://www.yr.no/place/Canada/Québec/Mont_Wapaskouch

https://www.yr.no/place/Canada/Québec/Pic_Longview/

(most recently viewed 2 June 2020)

28 CERTIFICATE OF AUTHORS

28.1 Paul Daigle, géo.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: “NI 43-101 Feasibility Study Troilus Gold – Copper Project, Québec Canada” (the Technical Report) dated 28 June 2024, with an effective date of 14 May 2024.

I, Paul Daigle, géo., do hereby certify that:

- I am a Principal Resource Geologist with AGP Mining Consultants Inc., with a business address at #246-132K Commerce Park Dr., Barrie ON L4N 0Z7 Canada.
- I am a graduate of Concordia University, Montreal, Canada (B.Sc. Geology) in 1989.
- I am a member in good standing of the Ordre des géologues du Québec (No. 1632).
- I have practiced my profession in the mining industry continuously since graduation. My relevant experience includes over 33 years in the mining sector in the exploration and diamond drill programs, managing data, and estimating resources. I have been involved in numerous precious metal projects in similar precious metal deposits within Archean/Proterozoic greenstone belts. My most recent experience includes the Detour Gold Deposit, Canada, and the Boto Gold Project, Senegal.
- I have read the definition of “qualified person” set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) and certify that by virtue of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a “qualified person” for the purposes of NI 43-101.
- I am independent of the issuer, Troilus Gold Corp., as defined in Section 1.5 of NI 43-101.
- I am responsible for Sections 1.1 to 1.10, 1.12, 4 to 12, 14, 23, 25.1, and 26.1 of this report and accept professional responsibility for those sections of the Technical Report.
- I have had prior involvement with the Troilus Gold Project that is the subject of the Technical Report. I was involved with the “Technical Report and Mineral Resource Estimate on the Troilus Gold-Copper Project, Québec, Canada” (27 August 2020) and the “Preliminary Economic Assessment of the Troilus Gold Project, Québec, Canada” (14 October 2020).
- My most recent site visit to the Troilus Gold Project described in this report was from October 5 to 7, 2022 for two days
- As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
- I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 28th day of June 2024, in Toronto, Ontario, Canada.

“signed electronically”

Paul Daigle, géo.

28.2 Marc Rougier, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Marc Rougier, certify that:

(a) I am a Principal Geotechnical Engineer at:

WSP Canada Inc.
6925 Century Ave
Mississauga, Ontario
L5N 0E3 Canada

(b) This certificate applies to the technical report titled NI 43-101 Feasibility Study Troilus Gold - Copper Project, Québec Canada with an effective date of: 14 May 2024 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of Queen's University with a BSc in Geological Engineering, 1991. I am a member the Ordre des Ingénieurs du Québec (Member ID 5055618). My relevant experience after graduation and over 30 years for the purpose of the Technical Report includes open pit geotechnical engineering and slope design, mining geotechnical engineering and physical hydrogeology for studies, operations and closure.

(d) My most recent personal inspection of each property described in the Technical Report occurred on November 9 to 11, 2020 and was for a duration of 2 days.

(e) I am responsible for Sections 1.14.1, 2.3.2, 16.2, 16.3.10, 16.3.11, 25.2 and 26.2 of the Technical Report.

(f) I am independent of Troilus Gold Corp. as described in section 1.5 of NI 43-101.

(g) I have not had prior involvement with the property that is the subject of the Technical Report.

(h) I have read NI 43-101 and the part of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Oakville, Ontario this June 28 of June 2024.

"signed electronically"

Marc Rougier, P.Eng., (OIQ Member ID 5055618)

28.3 Ryda Peung, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Feasibility Study Troilus Gold – Copper Project, Québec Canada" (the Technical Report) dated 28 June 2024, with an effective date of 14 May 2024.

I, Ryda Peung, P.Eng. do hereby certify that:

- I am a Principal Process Engineer with Lycopodium Minerals Canada Ltd., with a business address at 5090 Explorer Drive, Suite 700, Mississauga, ON L4W 4T9, Canada.
- I am a graduate of the University of Waterloo with a Bachelor of Applied Science degree, honours Chemical Engineering, 2008.
- I am a member in good standing with the Ontario Professional Engineers (Member Number 100136514).
- I am a member with the Ordre des Ingénieurs du Québec (Member Number 6065166).
- I have practiced my profession in the mining and metals industry continuously since graduation. My relevant experience includes over 16 years of design and engineering of minerals processing plants, with expertise in gold processing.
- I have read the definition of "qualified person" set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" for the purposes of NI 43-101.
- I am independent of the issuer, Troilus Gold Corp., as defined in Section 1.5 of NI 43-101.
- I am responsible for Sections 1.11, 1.15, 13, 17, 21.2, 25.4, and 26.4 of this report and accept professional responsibility for those sections of the Technical Report.
- I have not had any previous involvement with the Project.
- I have not visited the Troilus project site.
- As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
- I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible for, have been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 28th day of June 2024, in Toronto, Ontario, Canada.

"signed electronically"

Ryda Peung, P.Eng.

28.4 Willie Hamilton, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Feasibility Study Troilus Gold – Copper Project, Québec Canada" (the Technical Report) dated 28 June 2024, with an effective date of 14 May 2024.

I, Willie Hamilton, P.Eng., do hereby certify that:

- I am a Principal Mine Engineer with AGP Mining Consultants Inc., with a business address at #246-132K Commerce Park Dr., Barrie ON L4N 0Z7, Canada.
- I am a graduate of the University of Alberta with a Bachelor of Science in Mining Engineering in 1988 and a Masters of Science in Mining Engineering in 1990.
- I am a member in good standing of the Ordre des ingénieurs du Québec (No. 6067531).
- I have practiced my profession in the mining industry continuously since graduation. My relevant experience includes 34 years in operations and consulting at open-pit and underground, hard and soft-rock mines in Canada and the United States. I have expertise in mine planning, scheduling, pit optimization, and mineral reserve calculations, as well as project evaluation work for all sizes of studies.
- I have read the definition of "qualified person" set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" for the purposes of NI 43-101.
- I am independent of the issuer, Troilus Gold Corp., as defined in Section 1.5 of NI 43-101.
- I am responsible for Sections 1.13, 1.14(.2), 15, 16.1, 16.3(.1 to .9), 25.3, and 26.3 of this report and accept professional responsibility for those sections of the Technical Report.
- I have had previous involvement with the Project that is the subject of the Report. I was involved with the "Preliminary Economic Assessment of the Troilus Gold Project, Quebec, Canada" (14 October 2020).
- I have not visited the Troilus project site.
- As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
- I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 28th day of June 2024, in Calgary, Alberta, Canada.

"signed electronically"

Willie Hamilton, P.Eng.

28.5 Zunedbhai Shaikh, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Feasibility Study Troilus Gold – Copper Project, Québec Canada" (the Technical Report) dated 28 June 2024, with an effective date of 14 May 2024.

I, Zunedbhai Shaikh, P.Eng., do hereby certify that:

- I am a Senior Mechanical Engineer with Lycopodium Minerals Canada Ltd., with a business address at 5090 Explorer Drive, Suite 700, Mississauga, ON L4W 4T9 Canada.
- I am a graduate of the Toronto Metropolitan University (formerly Ryerson University) with a Bachelor of Engineering, 2008.
- I am a member in good standing with the Ontario Professional Engineers (Member Number 100137621) and the Ordre des Ingénieurs du Québec (Member Number 6065543).
- I have practiced my profession in the mining and metals industry continuously since graduation. My relevant experience includes over 16 years of mechanical and piping design of minerals processing plants and associated infrastructure.
- I have read the definition of "qualified person" set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" for the purposes of NI 43-101.
- I am independent of the issuer, Troilus Gold Corp., as defined in Section 1.5 of NI 43-101.
- I am responsible for Sections 1.15, 1.16, 18.1 to 18.9, 18.11 to 18.14, 25.5(.1), and 26.5(.1) of this report and accept professional responsibility for those sections of the Technical Report.
- I have not had any previous involvement with the Project as an independent Qualified Person.
- My most recent site visit to the Troilus Gold Project described in the Technical Report from September 5 to 7, 2022 for two days
- As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible for, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101 and Form 43-101F1.

Dated this 28th day of June 2024, in Toronto, Ontario, Canada.

"signed electronically"

Zunedbhai Shaikh, P.Eng.

28.6 Laurent Gareau, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Laurent Gareau, certify that:

(a) I am a Principal Geotechnical Engineer at:

WSP Canada Inc.
6925 Century Ave
Mississauga, Ontario
L5N 0E3 Canada

(b) This certificate applies to the technical report titled NI 43-101 Feasibility Study Troilus Gold - Copper Project, Québec Canada with an effective date of: 14 May 2024 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of Queen's University with a BSc in Geological Engineering, 1991. I am a member the Ordre des Ingénieurs du Québec (Member ID 5092650). My relevant experience after graduation and over 30 years for the purpose of the Technical Report includes open pit geotechnical engineering and slope design, mining geotechnical engineering and physical hydrogeology for studies, operations and closure.

(d) My most recent personal inspection of each property described in the Technical Report occurred on September 6 to 7, 2023 and was for a duration of 2 days.

(e) I am responsible for Sections 1.22.6 (TMF), 18.16.1 to 18.16.5, 18.16.7, 25.5.3 and 26.5.3 of the Technical Report.

(f) I am independent of Troilus Gold Corp. as described in section 1.5 of NI 43-101.

(g) I have not had prior involvement with the property that is the subject of the Technical Report.

(h) I have read NI 43-101 and the part of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Oakville, Ontario this June 28 of June 2024.

"signed electronically"

Laurent Gareau, P.Eng., (OIQ Member ID 5092650)

28.7 Vlad Rojanschi, , P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Vlad Rojanschi certify that:

(a) I am a Senior Water Resources Engineer at:

WSP Canada Inc.
237 4th Ave SW, Suite 3300
Calgary, Alberta,
T2P 4K3 Canada

(b) This certificate applies to the technical report titled "NI 43-101 Feasibility Study Troilus Gold - Copper Project, Québec Canada" with an effective date of: 14 May 2024 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of the Technical University for Civil Engineering Bucharest (Bachler of Engineering) and of the Stuttgart University (Master of Science and Dr.-Ing. – Ph.D. equivalent) in the areas of water resources engineering and management and hydraulic engineering. I am a member of the Ordre des ingénieurs du Québec (Member ID 5000611). I have more than 15 years of experience in the specific area of mine water management and in associated hydrology and hydraulic studies and design.

(d) My most recent personal inspection of each property described in the Technical Report occurred on September 6 to 7, 2023 and was for a duration of 2 days.

(e) I am responsible for Sections 1.22.6 (SWWM), 16.3.12,18.15, 18.16.6, 25.5.2, and 26.5.2 of the Technical Report.

(f) I am independent of Troilus Gold Corp. as described in section 1.5 of NI 43-101.

(g) I have not had prior involvement with the property that is the subject of the Technical Report.

(h) I have read NI 43-101 and the part of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Oakville, Ontario this June 28 of June 2024.

"signed electronically"

Vlad Rojanschi, P.Eng., (OIQ Member ID 5000611)

28.8 Pierre Primeau, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Pierre Primeau, certify that:

(a) I am a Principal Process Engineer at:

WSP Canada Inc.
33 Mackenzie Street, Suite 100
Sudbury, Ontario
P3C 4Y1 Canada

(b) This certificate applies to the technical report titled “NI 43-101 Feasibility Study Troilus Gold - Copper Project, Québec Canada” with an effective date of: 14 May 2024 (the “Technical Report”).

(c) I am a “qualified person” for the purposes of National Instrument 43-101 (“NI 43-101”). My qualifications as a qualified person are as follows. I am a graduate of Laurentian University. I am a member the Ordre des Ingénieurs du Québec (Member ID 111138). My relevant experience after graduation and over 35 years for the purpose of the Technical Report includes the configurations and designs of tailings dewatering and treatment systems including pipeline transport systems from conceptual to feasibility design.

(d) The requirement for a site visit is not applicable to me.

(e) I am responsible for Sections 1.22.6 (WT), 18.15.7, 18.15.8, 25.5.4 and 26.5.4 of the Technical Report.

(f) I am independent of Troilus Gold Corp. as described in section 1.5 of NI 43-101.

(g) I have not had prior involvement with the property that is the subject of the Technical Report.

(h) I have read NI 43-101 and the part of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Sudbury, Ontario this 28th day of June 2024.

“signed electronically”

Pierre Primeau, P.Eng., (OIQ Member ID 111138)

28.9 Ann Lamontagne, Eng., Ph.D.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Feasibility Study Troilus Gold – Copper Project, Québec Canada" (the Technical Report) dated 28 June 2024, with an effective date of 14 May 2024.

I, Ann Lamontagne, Eng., Ph.D., do hereby certify that:

- I am an Engineer with Lamont Inc., with a business address at 2200, chemin de la Sagamité Québec, Qc, G2N 0B7.
- I am a graduate of the Laval University with a degree in civil engineering in 1990 and a PhD in mining environment in 2001.
- I am a member in good standing of the Ordre des Ingénieurs du Québec [104345].
- I have practiced my profession in the Mining Environment industry continuously since graduation. My relevant experience includes over 23 years of working of many aspects of Environment and permitting for various Projects and Operations. I have read the definition of "qualified person" set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" for the purposes of NI 43-101.
- I am independent of the issuer, Troilus Gold Corp., as defined in Section 1.5 of NI 43-101.
- I am responsible for Sections 1.18, 20, 25.6, and 26.6 of the Technical Report and accept professional responsibility for this section of the Technical Report.
- I have been involved [on the dewatering impact statement for Troilus from 2019 to 2021 and is a collaborator for production of the current Impact Study.
- My most recent site visit to the Site was October 5 to 7, 2022 for two days.
- As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
- I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible have been prepared in accordance with NI 43-101 and Form 43-101F1.

Dated this 28th day of June 2024, in Québec, Canada.

"signed electronically"

Ann Lamontagne, Eng., Ph.D.

28.10 Gordon Zurowski, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Feasibility Study Troilus Gold – Copper Project, Québec Canada" (the Technical Report) dated 28 June 2024, with an effective date of 14 May 2024.

I, Gordon Zurowski, P.Eng., do hereby certify that:

- I am a Principal Mine Engineer with AGP Mining Consultants Inc., with a business address at #246-132K Commerce Park Dr., Barrie ON L4N 0Z7, Canada.
- I am a graduate of the University of Saskatchewan with a B.Sc. in in Geological Engineering in 1988.
- I am a member in good standing of the Professional Engineers of Ontario (#100077750).
- I have practiced my profession in the mining industry continuously since graduation. My relevant experience includes 30 years in mineral resource and reserve estimations and feasibility studies in Canada, the United States, Central and South America, Europe, Asia, Africa, and Australia.
- I have read the definition of "qualified person" set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" for the purposes of NI 43-101.
- I am independent of the issuer, Troilus Gold Corp., as defined in Section 1.5 of NI 43-101.
- I am responsible for Sections 1.1, 1.17, 1.19(.1 to. .2), 1.20 to 1.22, 2, 3, 18.10, 19, 21.1(.1), 21.1(4), 21.2(.1), 21.2(.2), 21.2(.4), 21.2(.5), 22, 24, 25.7, 25.8; and co-author of Sections 21.2(.4), and 21.2(.5) of this report and accept professional responsibility for those sections of the Technical Report.
- I have had prior involvement with the Troilus Gold Project that is the subject of the Technical Report. I was involved with the "Technical Report and Mineral Resource Estimate on the Troilus Gold-Copper Project, Québec, Canada" (27 August 2020) and the "Preliminary Economic Assessment of the Troilus Gold Project, Québec, Canada" (14 October 2020).
- My most recent site visit to the Troilus Gold Project described in this report was from July 13 to 15, 2020 for three days
- As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
- I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible have been prepared in accordance with NI 43-101 and Form 43-101F1.

Dated this 28th day of June 2024, in Toronto, Ontario, Canada.

"signed electronically"

Gordon Zurowski, P.Eng.

28.11 Balvinder Singh, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Feasibility Study Troilus Gold – Copper Project, Québec Canada" (the Technical Report) dated 28 June 2024, with an effective date of 14 May 2024.

I, Balvinder Singh, P.Eng., do hereby certify that:

- I am a Senior Director of Projects with Lycopodium Minerals Canada Ltd., with a business address at 5090 Explorer Drive, Suite 700, Mississauga, ON L4W 4T9, Canada.
- I am a graduate of a Bachelor of Mechanical Engineering degree from PTU, India 2000 and a Master of Engineering from Anna University, India, 2002. I am also graduate of Master of Business Administration from Queen's University, ON, Canada, in 2011.
- I am a member in good standing of the Ontario Professional Engineers (Member Number 100136514), Association of Professional Engineers and Geoscientists of Alberta (Member Number 137615), and Ordre des Ingenieurs du Québec (Temp Member Number 6065629).
- I have practiced my profession for over 20 years since graduation. I have been directly involved in performing and overseeing studies and project delivery of processing plants as Study Manager, Project Manager and Study/Project Sponsor.
- I have read the definition of "qualified person" set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" for the purposes of NI 43-101.
- I am independent of the issuer, Troilus Gold Corp., as defined in Section 1.5 of NI 43-101.
- I am responsible for Sections 21.1(.2), 21.1(.3), 21.1(.5 to .11) of this report and accept professional responsibility for those sections of the Technical Report.
- I have not had any previous involvement with the Project.
- I have not visited the Troilus project site.
- As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible for, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
- I have read NI 43-101 and Form 43-101F1, and the parts of the Technical Report for which I am responsible have been prepared in accordance with NI 43-101 and Form 43-101F1.

Dated this 28th day of June 2024, in Toronto, Ontario, Canada.

"signed electronically"

Balvinder Singh, P.Eng.

APPENDIX A – METALLURGICAL TESTWORK

ABA TESTWORK

BASE MET REPORTS

ERIEZ REPORTS

FLS REPORTS

KCA & HAZEN REPORTS

OLD TESTWORK REPORTS

APPENDIX B – WATER MANAGEMENT

APPENDIX C – PROCESS

OMC REPORT

PROCESS DESIGN CRITERIA (PDC) & MASS AND WATER BALANCE

PFDS

PROCESS CONTROL DIAGRAMS (P&IDS)

HAZID

Mechanical

Piping

Concrete and Structural

ELECTRICAL AND INSTRUMENTATION

APPENDIX D – TAILINGS

APPENDIX E – MINING

APPENDIX F – CAPEX

Appendix G – OPEX

APPENDIX H – ECONOMIC MODEL

APPENDIX I – PROJECT SCHEDULE

APPENDIX J – RISK REGISTER

APPENDIX K – LOGISTICS STUDY