TROILUS

Preliminary Economic Assessment of the Troilus Gold Project Quebec, Canada

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Prepared by

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(Note: Canadian currency used throughout unless otherwise noted)

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Units of Measure

Above mean sea level	amsl
Acre	ac
Ampere	A
Annum (year)	a
Billion	B
Billion tonnes	Bt
Billion years ago	Ga
British thermal unit	BTU
Centimetre	cm
Cubic centimetre	cm3
Cubic feet per minute	cfm
Cubic feet per second	ft3/s
Cubic foot	ft3
Cubic inch	in3
Cubic metre	m3
Cubic yard	yd3
Coefficients of Variation	CVs
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Dead weight tonnes	DWT
Decibel adjusted	dBa
Decibel	dB
Degree	°
Degrees Celsius	°C
Diameter	ø
Dollar (American)	US$
Dollar (Canadian)	CDN$
Dry metric ton	dmt
Foot	ft
Gallon	gal
Gallons per minute (US)	gpm
Gigajoule	GJ
Gigapascal	GPa
Gigawatt	GW
Gram	g
Grams per litre	g/L
Grams per tonne	g/t
Greater than	>

Hectare (10,000 m2)	ha
Hertz	Hz
Horsepower	hp
Hour	h
Hours per day	h/d
Hours per week	h/wk
Hours per year	h/a
Inch	"
Kilo (thousand)	k
Kilogram	kg
Kilograms per cubic metre	kg/m3
Kilograms per hour	kg/h
Kilograms per square metre	kg/m2
Kilometre	km
Kilometres per hour	km/h
Kilopascal	kPa
Kilotonne	kt
Kilovolt	kV
Kilovolt-ampere	kVA
Kilovolts	kV
Kilowatt	kW
Kilowatt hour	kWh
Kilowatt hours per tonne (metric ton)	kWh/t
Kilowatt hours per year	kWh/a
Less than	<
Litre	L
Litres per minute	L/min
Megabytes per second	Mb/sec
Megapascal	MPa
Megavolt-ampere	MVA
Megawatt	MW
Metre	m
Metres above sea level	masl
Metres Baltic sea level	mbsl
Metres per minute	m/min
Metres per second	m/s
Metric ton (tonne)	t
Microns	µm
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimetre	mm
Million	M
Million bank cubic metres	Mbm3

Million tonnes	Mt
Minute (plane angle)	'
Minute (time)	min
Month	mo
Ounce	oz
Pascal	Pa
Centipoise	mPa·s
Parts per million	ppm
Parts per billion	ppb
Percent	%
Pound(s)	lb
Pounds per square inch	psi
Revolutions per minute	rpm
Second (plane angle)	"
Second (time)	sec
Specific gravity	SG
Square centimetre	cm2
Square foot	ft2
Square inch	in2
Square kilometre	km2
Square metre	m2
Thousand tonnes	kt
Three Dimensional	3D
Tonne (1,000 kg)	t
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
Tonnes seconds per hour metre cubed	ts/hm3
Total	T
Volt	V
Week	wk
Weight/weight	w/w
Wet metric ton	wmt

Abbreviations and Acronyms

Absolute Relative Difference	ABRD
Acid Base Accounting	ABA
Acid Rock Drainage	ARD
Alpine Tundra	AT
Atomic Absorption Spectrophotometer	AAS
Atomic Absorption	AA
British Columbia Environmental Assessment Act	BCEAA
British Columbia Environmental Assessment Office	BCEAO
British Columbia Environmental Assessment	BCEA
British Columbia	BC
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
Carbon-in-leach	CIL
Caterpillar's® Fleet Production and Cost Analysis software	FPC
Closed-circuit Television	CCTV
Coefficient of Variation	CV
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 equivalent	AuEq
Heating, Ventilating, and Air Conditioning	HVAC
High Pressure Grinding Rolls	HPGR
Indicator Kriging	IK
Inductively Coupled Plasma Atomic Emission Spectroscopy	ICP-AES
Inductively Coupled Plasma	ICP
Inspectorate America Corp.	Inspectorate
Interior Cedar – Hemlock	ICH
Internal rate of return	IRR
International Congress on Large Dams	ICOLD
Inverse Distance Cubed	ID3

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 North	mN
Mineral Deposits Research Unit	MDRU
Mineral Titles Online	MTO
National Instrument 43-101	NI 43-101
Nearest Neighbour	NN
Net Invoice Value	NIV
Net Present Value	NPV
Net Smelter Prices	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 Persons	QPs
Quality assurance	QA
Quality control	QC
Rhenium	Re
Rock Mass Rating	RMR '76
Rock Quality Designation	RQD
SAG Mill/Ball Mill/Pebble Crushing	SABC
Semi-autogenous Grinding	SAG
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
Workplace Hazardous Materials Information System	WHMIS
X-Ray Fluorescence Spectrometer	XRF

Forward Looking Statements and Cautionary Language

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that the Indicated Mineral Resources will be converted to the Probable Mineral Reserve category, and there is no certainty that the updated Mineral Resource statement will be realized.

The PEA is preliminary in nature, includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

This Technical Report contains “forward-looking statements” within the meaning of applicable Canadian securities legislation. Forward-looking statements include, but are not limited to, the results of the PEA, statements regarding the impact and implications of the economic statements related to the PEA, such as future projected production, costs, including without limitation, AISC, total cash costs, cash costs per ounce, capital costs and operating costs, statements with respect to Mineral Resource estimates, recovery rates, IRR, NPV, mine life, CAPEX, payback period, sensitivity analysis to gold prices, timing of future studies including the pre-feasibility study, environmental assessments and development plans; the development potential and timetable of the project; the estimation of mineral resources; realization of mineral resource estimates; the timing and amount of estimated future exploration; costs of future activities; capital and operating expenditures; success of exploration activities; support from local communities;. 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”, “contemplates”, “goal”, “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”. Forward-looking statements are made based upon certain assumptions and other important facts that, if untrue, could cause the actual results, performances, 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 regarding present and future business strategies and the environment in which Troilus will operate in the future. Certain important factors that could cause actual results, performances, or achievements to differ materially from those in the forward-looking statements include, amongst others, currency fluctuations, the global economic climate, dilution, share price volatility and competition. 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: the impact the COVID 19 pandemic may have on the Company’s activities (including without limitation on its employees and suppliers) and the economy in general; the impact of the recovery post COVID 19 pandemic and its impact on gold and other metals; there being no assurance that the exploration program or programs of the Company will result in expanded mineral resources; risks and uncertainties inherent to mineral resource estimates; the high degree of uncertainties inherent to preliminary economic assessments and other mining and economic studies which are based to a significant extent on various assumptions; variations in gold prices and other precious metals, exchange rate fluctuations; variations in cost of supplies and labour; receipt of necessary approvals; 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 latest annual information form of the Company, and in other continuous disclosure documents of the Company available under the Company's profile at www.sedar.com.

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Mineral resource estimates have been prepared in accordance with the requirements of Canadian securities laws, which differ from the requirements of U.S. securities laws. The terms "mineral resource", "measured mineral resource", "indicated mineral resource" and "inferred mineral resource" are defined in NI 43-101 and recognized by Canadian securities laws but are not defined terms or recognized under U.S. securities laws. U.S. investors are cautioned not to assume that any part or all of mineral deposits in these categories will ever be upgraded to mineral reserves. "Inferred mineral resources" have a great amount of uncertainty as to their existence, and great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an "inferred mineral resource" will ever be upgraded to a higher category. Under Canadian securities laws, estimates of "inferred mineral resources" may not form the basis of feasibility or pre-feasibility studies. U.S. investors are cautioned not to assume that all or any part of an inferred mineral resource exists or is economically or legally mineable. Accordingly, these mineral resource estimates and related information may not be comparable to similar information made public by U.S. companies subject to the reporting and disclosure requirements under the U.S. federal securities laws and the rules and regulations thereunder.

Non-IFRS Financial Measures

The Company has included certain non-IFRS financial measures in this news release, such as Initial Capital Cost, Cash Operating Costs ,Total Cash Cost, All-In Sustaining Cost, Expansion Capital, and Capital Intensity, 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.

Non-IFRS financial measures used in this news release and common to the gold mining industry are defined below.

Total Cash Costs and Total Cash Costs per Ounce

Total Cash Costs are reflective of the cost of production. Total Cash Costs reported in the PEA include mining costs, processing & water treatment costs, general and administrative costs of the mine, off-site costs, refining costs, transportation costs and royalties. Total Cash Costs per Ounce is calculated as Total Cash Costs divided by payable gold ounces.

All-in Sustaining Costs ("AISC") and AISC per Ounce

AISC is reflective of all of the expenditures that are required to produce an ounce of gold from operations. AISC reported in the PEAS includes total cash costs, sustaining capital, expansion capital and closure costs, but excludes corporate general and administrative costs and salvage. AISC per Ounce is calculated as AISC divided by payable gold ounces.

1 SUMMARY

1.1 Introduction

Troilus Gold Corp. (Troilus) is a Canadian exploration and development company, based in Toronto, Canada, and is publicly-listed on the Toronto Stock Exchange (TSX). Troilus is focused on the development of the Troilus Gold Project, which includes the historic Troilus Mine; and the Troilus Frotêt Project. Troilus holds a 100% interest in the mineral rights for the Projects.

The Troilus Mine was an open pit operation producing gold, copper, and silver continuously from November 1996 to April 2009. The Troilus Mine produced over two million ounces (oz) of gold and approximately 70,000 tonnes (t) of copper. After the mine ceased production in 2009, the 20,000 tonnes per day (tpd) mill processed low grade stockpiles until June 29, 2010. Following this, the mill was sold and shipped to Mexico and the main camp facilities were dismantled in late 2010.

This Technical Report and Preliminary Economic Assessment (PEA) was prepared by AGP for Troilus to present the results of a preliminary economic evaluation of Troilus Gold Project. This Technical Report was prepared for the Troilus Property in accordance with NI 43-101 and Form 43-101F1. The mineral resources used in the PEA were prepared on the Z87, J4/J5, and Southwest (SW) Zone within the Troilus Gold Project.

The PEA concluded that the Troilus Gold Project could be developed as a phased open pit and underground operation. The pits and underground would feed a 35,000 tpd mill with a total of 192.5 Mt of mill feed grading 0.71 gpt gold, 0.08% copper and 0.97 gpt silver over a 21 year mine life. One year of pre-stripping activity would be required. The process facility would be a conventional crushing and grinding plant followed by flotation, to produce a gold rich copper concentrate. Some of the gold would be recovered by gravity to produce a dore on site. A site layout illustrating the proposed location of required infrastructure, mining and processing facilities is shown in Figure 1-1.

At a gold price of $US 1,475/oz, copper price of $US 3.00/lb, silver price of $US 20/oz and an exchange rate of 0.74 (CDN:US) the project is estimated to have an after-tax IRR of 22.9% and a pay-back period of 4.0 years after start of production. At a discount rate of 5%, the after tax NPV is estimated at $CDN 778M ($US 576M).

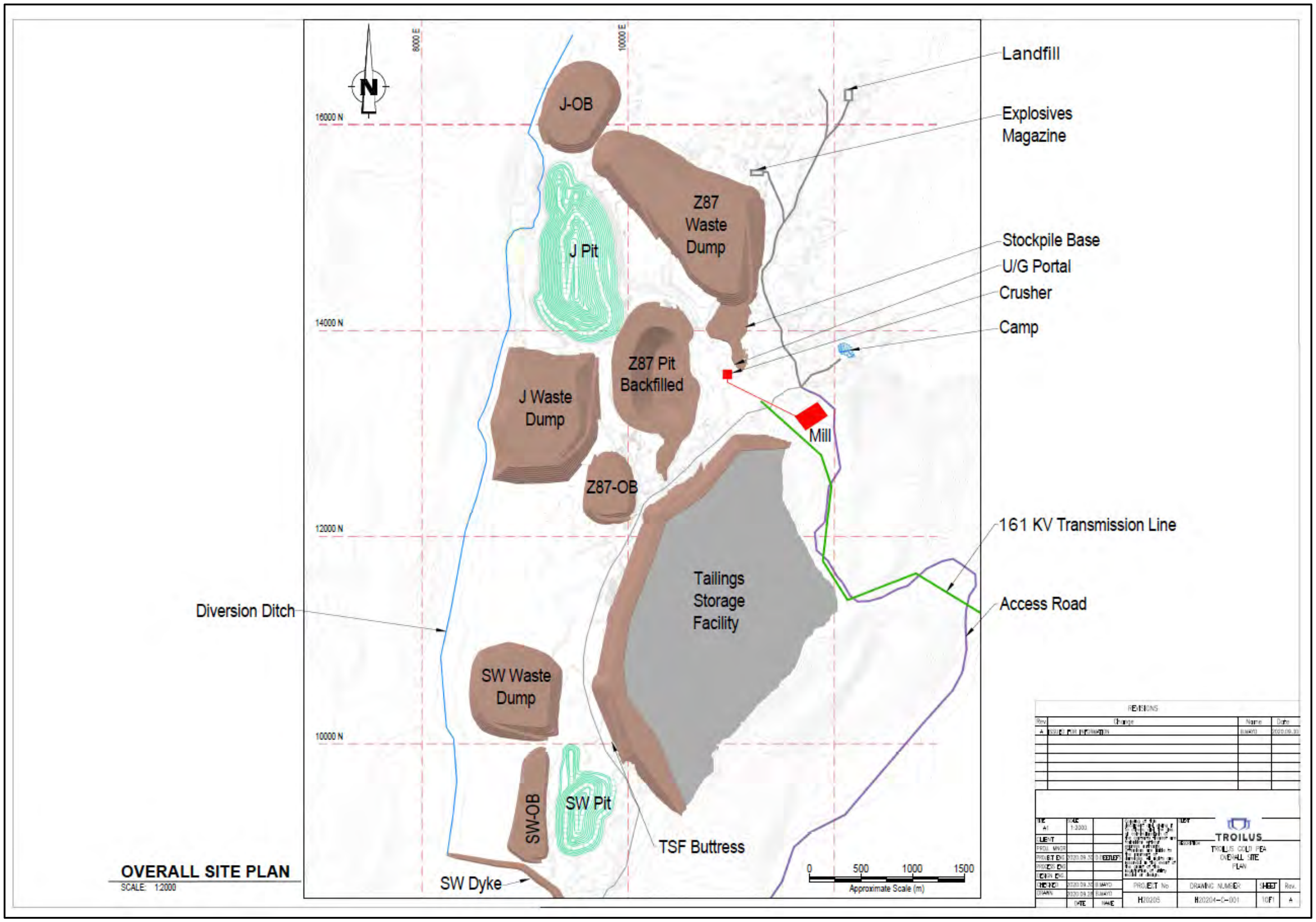
The life of mine capital cost for the project is estimated at $CDN 1,132M ($US 838.6M), with an initial capital expenditure of $CDN 449.5M ($US 333M) which includes $CDN 94.7($US 70.1M) pre-strip costs which are capitalized.

The PEA utilizes Indicated, and Inferred resources from the 87, J and SW zones for calculation of potential economics. The Indicated resource was 74.9% of the total mill feed. Inferred material, representing 25.1% of the mill feed, is included in the total mill feed. There is no certainty the assumptions utilized in the PEA will be realized. Inferred mineral resources are presently considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as reserves. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Based on the results of the PEA study, AGP recommends that Troilus Gold proceed with a Prefeasibility study as part of the project development plan to help determine a project execution decision. Recommendations and associated budgets are provided to ensure sufficient information is available going forward.

With the current level of information for the Project, AGP does not foresee any Mineral Resources, potential economics or environmental issues that would inhibit the Project from advancing to further levels of study.

Figure 1-1: Troilus Gold Project Site Layout



1.2 Property Location and Description

The Troilus Gold Project (the Project) is located in central Quebec and is situated approximately 120 km north of Chibougamau. The Troilus Gold Property is defined by the mineral rights that are 100% held by Troilus. The mineral rights for the Property are comprised of a single Mining Lease (Bail Minier), and 1,988 mineral claims (Titres Miniers) and covers a total area of approximately 107,321 ha. All mineral rights are in good standing.

1.3 History

Prior to 1985, the Project area was subject to regional exploration by Falconbridge Ltd. (now Glencore) and Selco Mining Corp. The Government of Quebec also conducted an airborne survey over 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 issued. 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.

1.4 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 Quebec. 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 Troilus Property occur around the margins of the Troilus Diorite, and comprise the Z87, Z87S, and the J zones (comprising J4 and J5). Other important mineralized zones discovered to date include the northern continuity of the J zones, 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 matrix of the diorite breccia, the diorite and the felsic dikes represent the main host rocks for the mineralized intervals.

1.5 Exploration and Drilling

Since the formation of Troilus, exploration activities have been focussed on exploration targets around the main Z87 and J4/J5 Zones. These targets included areas to the northeast of J4/J5 (J4N or Allongé, L'Ours, Carcajou), southwest of Z87 (Z87S, Z86, SW and Sand Pit)

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) was retained to conduct a structural geology investigation at the Project. The study focused on the exposed geology in the Z87 open pit and the J4/J5 open pit.

In June 2020, Troilus completed a preliminary field exploration program applying a new regional structural and geological model, developed over the last two years, to the recently expanded Troilus-Frôtet Property. This property is situated to the south of the main mineralized zones of Z87, J Zones and the SW Zone. A regional airborne geophysical survey was also completed. Initial results have led to the discovery of the Beyan Gold Zone, situated approximately 8 km southwest and along strike of the SW Zone of the Troilus Gold Project.

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 2020. Troilus completed 91 drill holes totalling 37,510 m in 2018; 75 drill holes totalling 35,685 m from 2019; and 17 drill holes totalling 6,037 in 2020. Most of the 2018 and 2019 drill holes targeted Z87 and the J zones at depth and along strike. In the SW Zone, 24 drill holes were completed, totalling 8,500 m.

The current resource drill hole database contains 829 drill holes totalling approximately 207,945 m where the majority of the drilling targeted Z87, J4/J5 and SW Zones and includes 69 exploration drill holes.

1.6 Mineral Resources

The Mineral Resources for the Project include the three principal mineralized zones: Z87, J4/J5 and SW Zones. The mineral resources were prepared and disclosed in accordance with the CIM Definitions for Mineral Resources and Mineral Reserves (2014). The QP responsible for the resource estimates is Mr. Paul Daigle, géo., Associate Resource Geologist for AGP. The effective date of these mineral resource is 20 July 2020.

The mineral resources were prepared using interpreted mineralized domains at each of the three zones. The resource estimates were completed using Geovia GEMS™ 6.8.3 resource estimation software. The blocks model grades were estimated using ordinary kriging interpolation method using 2m capped composites.

Table 1-1 presents a summary of the mineral resources for the Project.

Table 1-1: Mineral Resources for the Troilus Project; combined open pit and underground resources

Classification	Tonnes (,000t)	Grade				Contained Metal			
		Au (gpt Au)	Cu (% Cu)	Ag (gpt Ag)	AuEQ (gpt AuEQ)	Au (Moz)	Cu (Mlbs)	Ag (Moz)	AuEQ (M oz)
Indicated	177.3	0.75	0.08	1.17	0.87	4.30	322.60	6.66	4.96
Inferred	116.7	0.73	0.07	1.04	0.84	2.76	189.73	3.91	3.15

Notes:

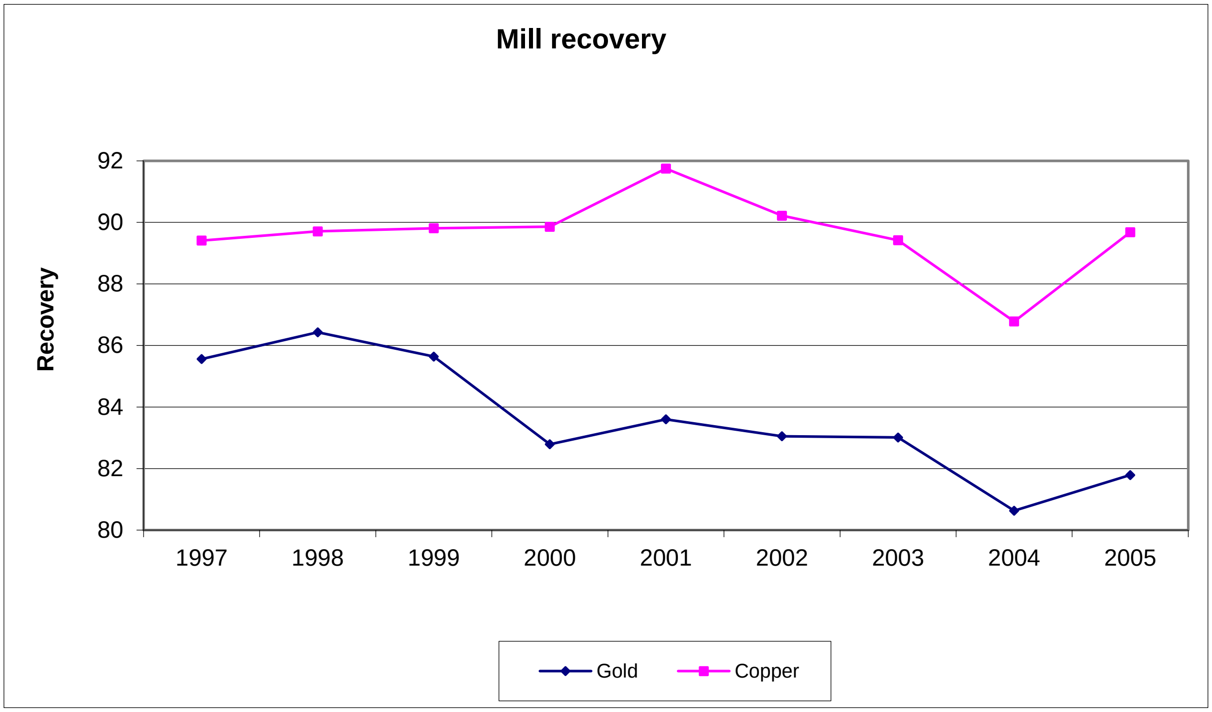
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- Summation errors may occur due to rounding.
- Open pit mineral resources are reported within optimized constraining shells.
- Open pit cut-off grade is 0.3 gpt AuEQ where the metal equivalents were calculated as follows:
  - Z87 Zone AuEq = Au grade + 1.2566 * Cu grade + 0.0103 * Ag grade
  - J4/J5 Zone AuEq = Au grade + 1.2979 * Cu grade + 0.0108 * Ag grade
  - SW Zone AuEq = Au grade + 1.2768 * Cu grade + 0.0106 * Ag grade
- Metal prices for the AuEQ formulas are: $US 1,600/ oz Au; $3.25/lb Cu, and $20.00/ oz Ag; with an exchange rate of US$1.00: CAD$1.25.
- Metal recoveries for the AuEQ formulas are:
  - Z87 Zone 83% for Au recovery, 92% for Cu recovery and 76% for Ag recovery
  - J4J5 Zone 82% for Au recovery, 88% for Cu recovery and 76% for Ag recovery
  - Z87 Zone 82.5% for Au recovery, 90% for Cu recovery and 76% for Ag recovery
- The resource constraining shells were generated with:
  - Metal Prices: Gold $US 1600/oz, Copper $US 3.25/lb, Silver $US 20/oz
  - Mining Costs:
    J Zone and 87 Zone base cost $Cdn 1.71/t moved,
    SW Zone base cost $Cdn 1.66/t moved
    Incremental cost $Cdn 0.03/t waste moved, $Cdn 0.02/t feed moved
  - Process and G&A Costs: $Cdn 8.44/t processed
  - Wall slopes: varied between 49.5 to 60 degrees depending on pit area and slope sector
  - Metal Recoveries:
    Gold: 90% all zones except in lower grade (Au < 1.2 g/t) portions of SW zone = 88%
    Copper: 90% all zones except in higher grade (Cu > 0.13%) portions of SW zone = 92%
    Silver: all zones 40%
- Underground cut-off grade is 0.9 AuEQ at Z87 Zone below constraining pit
- Capping of grades varied between 2.00 g/t Au and 26.00 g/t Au; between 1.00 g/t Ag and 20.00 g/t Ag on raw assays; and 1.00 %Cu on raw assays
- The density varies between 2.72 g/cm^3 and 2.91 g/cm^3 depending on mineralized zone.

1.7 Mineral Processing and Metallurgical Testing

Metallurgical testwork dating back to the early 1990's has been reviewed and is summarized within this report. A variety of samples from the different Troilus deposits have been tested, and the nature of samples is considered sufficiently diverse for preliminary metallurgical assessment. Older testwork (1990's) tended to focus on higher grade samples and thus additional work on lower grade material from within current mine plan areas is recommended for future studies.

Historical operations at the Troilus mine included a 10,000 tpd concentrator that utilized crushing, grinding gravity concentration and froth flotation to produce both a gold/silver doré and a copper/gold flotation concentrate. The plant was subsequently expanded to 18,000 tpd via a series of upgrades and a decision to coarsen the mill grind. Whilst this allowed for extra tonnage, the resultant drop in liberation had a negative impact on gold recovery, from approximately 86% to 82% in 2005 when operations were halted. A summary of results is shown in Figure 1-2

Figure 1-2: Historical Gold and Copper Mill Recoveries, Troilus Gold Mine



The results highlight the impact of grind on gold recovery, and hence within the flowsheet recommended by this study a finer grind (80% -75 µm) has been selected. This should help to obtain the 90% gold recovery targets used in the current work.

Metallurgical testwork completed between 1993 and 2020 includes, inter alia:

- Comminution work – including the determination of Abrasion index, Bond rod and ball mill index, and Drop Weight Test parameters.
- Laboratory scale testing of various froth flotation flowsheets (with and without gravity concentration testing).
- Laboratory scale testing of various cyanidation flowsheets (with and without gravity concentration testing). This work includes the evaluation of heap leaching with cyanide solutions.
- Pilot scale testing of flotation flowsheets.
- Laboratory scale testing of various “combination flowsheets” including both flotation and cyanidation. Cyanidation of flotation concentrate and/or tailing products, plus flotation of cyanidation residues have all been tested and reviewed.

According to the JKMRC database of projects, the A*b values for the four composites are low, which indicates that Troilus mineralization is in the “moderately hard” category and therefore will have relatively high resistance to breakage.

Various gravity testwork results indicate a range of gold recoveries, but in general it is felt that all Troilus samples have responded well to concentration using centrifugal type concentrators within the primary grinding circuit. In addition, the historical Troilus flowsheet included gravity recovery from a stream around the rougher concentrate regrind mill – utilizing higher g-force and targeting finer gravity recoverable gold. For this preliminary economic assessment, two stages of gravity concentration have been included in the flowsheet, and an average gravity gold recovery of 30% (to a doré product) has been assumed for all deposits.

Testwork indicates amenability to froth flotation for all samples. Concentrate grades and recoveries are variable and somewhat sensitive to head grade, but it is anticipated that the use of modern mineral processing equipment, including large scale tank cells, inert fine grinding mills (ISA Mill), and column flotation cells will achieve high copper and gold recoveries at saleable copper concentrate grades.

Samples are also amenable to gold recovery by cyanidation, with improved results achieved at finer grinds and after gravity recovery. Copper recovery is effectively zero for this processing route, however.

Combined flotation plus cyanidation flowsheets have been tested, and these show promise in terms of gold recovery. Both flotation of cyanide circuit residues (post-cyanide detox) and cyanidation of flotation circuit tailings have been trialed, with encouraging results. Additional testwork and economic evaluation would be required to fully assess these more complex flowsheets.

For this preliminary study, the metallurgical predictions given in Table 1-2 have been used as a basis for economic evaluation.

Table 1-2: Metallurgical Predictions for Flotation

Zone	Head Grade, % Cu	Conc Grade, % Cu	Recovery, %	Gold Recovery, %
87 Zone	0.09 average	23	90	60
J Zone	0.06 average	12	90	60
SW Zone	when >= 0.13	19	92	if Head Grade >1.2 g/t – 60 if Head Grade <1.2 g/t – 58
SW Zone	when < 0.13	17	90	if Head Grade >1.2 g/t – 60 if Head Grade <1.2 g/t – 58

Copper performance is related to head grade, and in cases where head grade is lower (<0.09% Cu) then reasonable copper concentrate grades become difficult to achieve without sacrificing copper and gold recovery. In these instances (J Zone for example) concentrate grade is sacrificed in return for recovery. This can be tolerated commercially as a result of the supplemental gold grades in these low grade copper products, and the fact that for several years it can be blended with higher grade products (87 UG zone or SW Zone).

The metallurgical performance of silver is not reported comprehensively in the metallurgical testwork, and therefore a somewhat conservative assumption of 40% recovery has been allowed across the board for this metal.

1.8 Mine Plan

1.8.1 Open Pit Mining

The PEA is based on the reactivation and expansion of the 87 and J Zone pits and the addition of a new area, the SW Zone. These pits provide the open pit feed material necessary to maintain the process plant feed rate at 35,000 t/d while operational.

The 87 pit is a single phase which provides 36.6 Mt of mill feed grading 0.72 gpt gold, 0.088% copper and 1.4 gpt silver for a gold equivalent grade of 0.85 gpt. Waste from this pit totals 149.4 Mt for a strip ratio of 4.1 (waste:mill feed). The 87 pit forms the top of the underground development.

The J zone pit has three phases. The phases total 94.8 Mt of mill feed grading 0.51 gpt gold, 0.06% copper, and 0.89 gpt silver for a gold equivalent grade of 0.60 gpt. Waste from the phases totaled 348.7 Mt for a strip ratio of 3.7:1 (waste:mill feed).

The new SW Zone pit is mined in two phases. They will produce 18.8 Mt of mill feed grading 0.64 gpt gold, 0.065% copper and 0.76 gpt silver for a gold equivalent grade of 0.74 gpt. The waste amounts to 93 Mt giving a strip ratio of 5.0:1 (waste:mill feed).

The phases are scheduled to provide 35,000 t/d of feed to the mill over a 14 year open pit mining life after one year of pre-production stripping. As the underground mine production comes online in Year 8 the open pit production drops to a level sufficient to keep the process plant at full capacity. The pits are sequenced to minimize initial stripping and provide higher feed grades in the early years of the mine life. This is accomplished with stockpiling of lower grade material which is used later in the mine life.

Initial mining starts in the 87 pit and the SW pit. These provide the highest grade to the mill early in the schedule. The 87 pit needs to be complete for the underground mine to produce material. The 87 pit finishes in Year 6. The other advantage of finishing the 87 pit early is that this can then be used for waste storage of material from the J pit.

The pits are built on 10 metre benches with safety berm placement each 20 metres. Inter-ramp angles vary from 47 to 53 degrees depending upon the wall orientation. Minimum mining widths of 60 metres were maintained in the design. Ramps are at maximum 10% gradient and vary in width from 25.5 m (single lane width) to 33.2 m (double lane width). They have been designed for 181 t haulage trucks.

The mine equipment fleet is anticipated to be financed to lower capital requirements. The fleet will be comprised of nine 200mm down the hole drills, two 22 m^3 hydraulic shovels and two 23 m^3 front end loaders. The truck fleet will total 28 trucks from Year 1 onwards. This is due to the long hauls from the SW pit, the tailings buttress buildup and the initial higher strip ratio reactivating the 87 and J pits. The usual assortment of dozers, graders, small backhoes, and other support equipment is considered in the equipment costing. A smaller front end loader (13 m^3) will be stationed at the primary crusher.

The waste dumps will be placed adjacent to the various pits. Waste from the 87 pit will be used to recontour and build upon the existing 87 waste dump. This will include wraparounds on the eastern side which will form the base for the low grade stockpile. The SW pit will develop a new waste dump to the west of the pit. The J pits will cover over an older facility to the south until the 87 pit is available for backfill. When that occurs, all remaining waste will be placed in the 87 pit from the J phases or on the tailings buttress. This allows the reclamation of the other facilities to be completed while mine operations are underway. A total of 292.1 Mm^3 has been designed and it is sufficient for the mine needs.

The LOM operating cost is estimated at $CDN 2.70/t of material mined. This includes equipment financing of $CDN 0.41/t of material mined.

Pre-production stripping costs of $CDN 94.7 million are capitalized. Initial mine equipment capital is $CDN 8.4 million with sustaining capital of $CDN 6.7 million.

1.8.2 Underground Mining

The development of the underground mine will commence once open pit production is established. Underground production will be mined concurrently with lower grade open pit material, thereby enhancing mill grade.

Inferred Resources account for 28% of the underground material to be processed. Only limited underground mine planning has previously been undertaken on the Z87 deposit.

The planned underground mining area is an extension of the Z87 deposit previously mined by open pit at Troilus. The depth of the existing open pit is now planned to be extended by open pit methods by around 50 m, to approximately 350 m below surface. The currently identified Measured, Indicated and Inferred Resources for the underground area extend to around 900 m below surface and measure a maximum of approximately 850 m along strike. The dip of the deposit varies from around 60^0 to around 40^0, averaging 55^0 in the north and central areas with the flatter dip to the south. An optimised in situ cut off grade of 0.8 equivalent g/t Au was calculated. Higher grade mineralized areas bifurcate in certain areas, but low grade intervening mineralization that allows for the mining of the full section from footwall to hangingwall at satisfactory grades was included in the study plans. Stopes vary in thickness up to 80 m true thickness with the thickness generally reducing with depth.

In general, ground conditions are considered to be good to very good with strong rock throughout the footwall, orebody and hanging wall sequences. Geological structure in the form of faults and low-angle, widely spaced joints have been identified in the exposed open pit sidewalls.

Trade off studies were undertaken that identified Slot and Mass Blast (S&MB) as the preferred mining method and Rail-Veyor as the preferred materials handling system.

S&MB will be the primary underground mining method used to exploit the Z87 deposit below the open pit floor and will provide 89% of the life of mine underground feed to the mill. The remaining 11% of underground mill feed will be mined using the sub level caving (SLC) method, which is located in the upper portion of the underground mining area, between the deepened open pit and the upper-most level of slot and mass blast stopes. Both of the selected mining methods - as well as the development and operation of the Rail-Veyor materials handling system - operate in a 'top-down' fashion, thus minimising and deferring the mine development necessary to place the mine in operation and sustain production over the life of mine. Initial production will be by SLC followed by S&MB.

Life of mine feed to the process plant is estimated to be 42.3Mt with an equivalent gold grade of 1.35 g/t at a steady-state production rate of 9,000 tpd. Underground mine capital expenditure is estimated to be $CDN 324M of development capital with $CDN 236 M sustaining capital during the remainder of mine life. Capital development will be undertaken by a contractor. Owner crews will undertake all subsequent mine activities apart from raising and deposit delineation. Life of mine underground operating costs are estimated to average $CDN 19.38 per tonne of process feed.

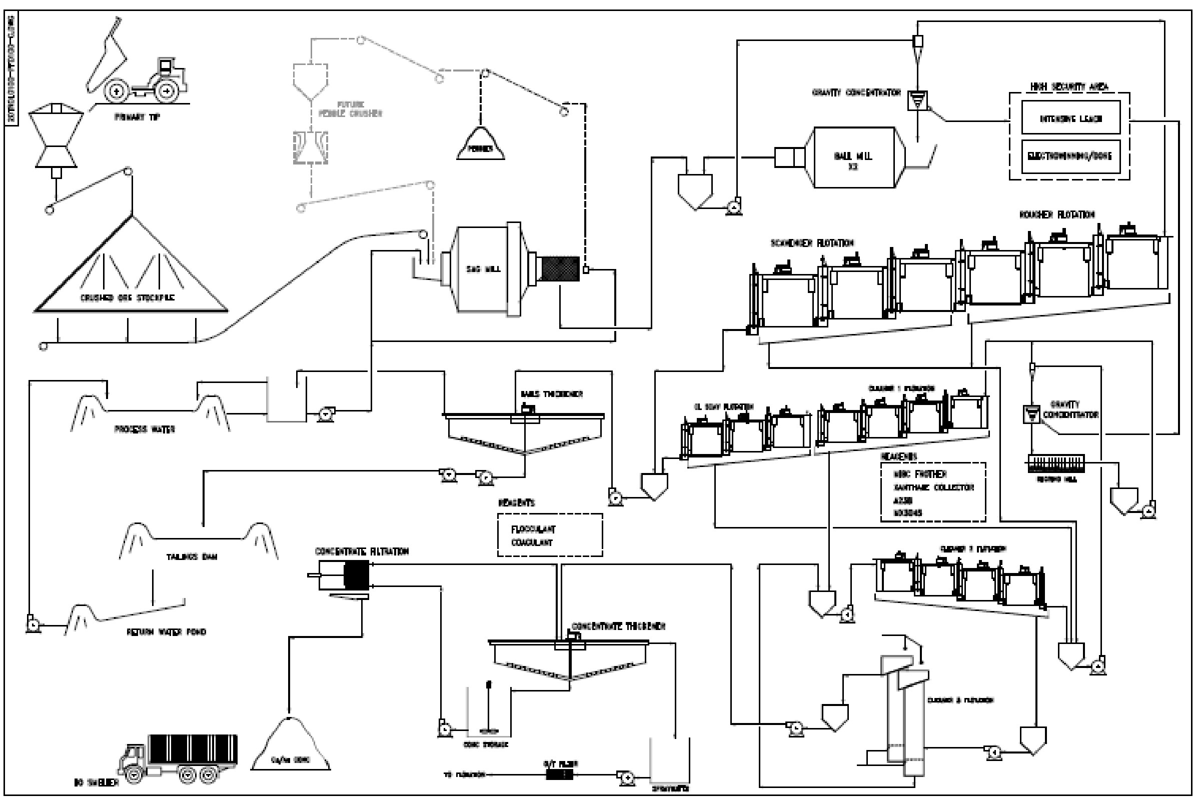
Large scale mobile equipment types were assumed to maximise productivity. The mobile equipment fleet will be financed.

1.9 Recovery Methods

As noted in Section 13, samples of Troilus mineralization have responded well to froth flotation and cyanidation after crushing, grinding and gravity concentration. Consideration of metallurgical performance, operating cost and anticipated revenues resulted in the selection of a standard multi-stage froth flotation circuit for this preliminary economic assessment.

The selected process plant has been sized for a throughput rate of 35,000 tpd. The flowsheet is illustrated in Figure 13-6 below.

Figure 1-3: Process Flowsheet



All samples of mineralization tested historically, plus the results of previous Troilus operations suggest that gravity recovery of gold will be economically beneficial and will result in a proportion of the gold product to be sold as a doré rather than a flotation concentrate. For this study, a conservative assumption of 30% gold recovery to doré has been used.

Grinding to a finer target of 80% -75 µm is anticipated to improve metallurgical performance and this will be achieved using a combination of SAG and ball milling. The flotation process will utilize modern large tank cell type flotation cells in combination with an inert fine grinding regrind mill to bring rougher concentrate to a size target of 80% -25 µm. Cleaner circuit equipment would include a mix of conventional tank cells and column flotation. Flotation concentrates would be dewatered using a thickener and a pressure filter to allow production of a filtered cake for transport to smelters.

Flotation tailing slurry would be thickened in a high rate tailing thickener and then pumped to the nearby tailing storage facility.

1.10 Infrastructure

The overall site plan is shown in Figure 1-1 and includes major facilities of the Troilus Project. This includes:

1. 87 Zone pit – only one phase in plan with the backfilled outline shown
2. J Zone pit – final phase of three planned for mining
3. SW Zone pit – final phase of two planned for mining
4. Primary crushing and conveying
5. Process plant location
6. Overburden and Waste Rock Storage facilities
7. No Name Creek diversion
8. Tailings Facility – at end of mine life with buttress waste storage facility

Access to the site will be provided by a new access road to the south and east of the tailings facility.

Grid power will be provided by the existing high voltage line to the current transformer substation. Existing electrical infrastructure includes the Hydro Quebec 161 MVA line to site. At site there are two 25MVA transformers in the current substation. This existing electrical infrastructure is sufficient for the PEA outlined requirements. Diesel backup power is also at the substation.

Raw water will be provided by an existing facility located in the lake north of the site and reclaim of water from the tailings facility

The existing No Name Creek diversion will be extended and realigned to provide access to waste storage facilities.

Waste rock and overburden will be stored in separate locations. The overburden material will be used for later revegetation of the waste facilities. Waste facilities will be actively reclaimed as they are constructed with dozers resloping to 26.5 degrees. This is to allow revegetation to occur as soon as possible.

A portion of the 87 Pit will be backfilled with waste material from the J zone phases. The entire pit is not backfilled due to timing of the various mining activities. The final interior portion will be resloped for reclaim purposes as well.

Tailings will be stored in the existing facility location. This facility will be expanded annually to accommodate the expected process tonnage. The facility has sufficient capacity to accommodate this tonnage. The material will be stored as a thickened tails with water reclaimed from the facility to offset process freshwater requirements. The facility will be expanded in a center line construction manner. Material for the dam will be provided by the mine which will bring additional material to buttress the facility.

Camp requirements are for 350 persons initially rising to 425 persons as the underground mine is established. The camp facilities are included as part of a quotation provided by a local vendor to supply all camp facilities and catering for the project life. The facilities will have accommodations, catering, lounges, and a fitness centre for Troilus personnel.

Water treatment facilities current exist at the tailings facility and J zone pit. The equipment at the J Zone pit is not required currently but would be expanded during operation to accommodate anticipated water pumping volumes. This expansion capital is included in the capital cost estimate.

Pit pumping requirements are estimated at 12,000 m^3 per day with a seasonal peak of 15,000 m^3 per day.

1.11 Environmental

The Troilus 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 Troilus site has currently two environmental statuses: exploration and closed(reclaimed) sites. The site has been reclaimed from the end of the previous operation, from 2011 to now. The waste piles and tailings pond have been revegetated The remaining work for closure is removing the pumps from the tailings and having the water flow naturally via a canal. The exploration status relates to the drilling and finding new resources for an eventual new operation.

In November 2019, the Company submitted an environmental impact study 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 Troilus property. The Company engaged in community consultations with impacted families on the Troilus 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 Baseline studies were conducted prior to the exploitation of the Mine in 1997 and due to the elapsed time, new baseline studies were undertaken by various consultants in 2019.

The baseline studies have and will continue to focus on a description of existing conditions, considering that the site has already been impacted by the operation of a mine for about 12 years, then has been partially restored.

No known environmental issues have been identified at the site that would materially affect the current mine, design, or scope of the needed environmental permits.

The diversion of the unnamed creek as proposed in the PEA will have to examined, as this will be the major environmental item for the Project.

The most substantive potential impacts of projects are generally associated with the long-term management of waste rock, tailings, mine water and process water and their downstream effects on water and fish habitat. As the project advances through the various stages of study, the application of appropriate engineering design, project planning, and implementation of responsible production and environmental management plans will mitigate any significant environmental effects.

The fact that the tailings area and waste piles have been on site since 1997 from the former mine with no significant environmental effects indicates that the risk of having issues with the same orebody is expected to be very low.

1.12 Markets

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

- Gold - $1,475 US/oz
- Copper - $3.00 US/lb
- Silver - $20.00 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 16.6% grade, the higher gold grade of 103 g/t will make it attractive to various worldwide smelters.

Handling considerations will be slightly different than normal concentrate in bulk as this is bagged to minimize the losses of gold during transportation.

Indicative smelter terms were provided to Troilus for the PEA 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. 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.

1.13 Capital and Operating Costs

1.13.1 Capital Costs

The initial and life of mine capital cost estimates for the Project are summarized in Table 1-3. All costs are expressed in Canadian Dollars (CDN) unless otherwise stated and are based on 2020 H1 2020 pricing. The mine capital costs consider full financing of the mine fleet which reduces the initial capital cost and transfers that to operating cost.

Table 1-3: Troilus Gold Project Capital Cost Estimate ($CDN)

Area	Initial Capital (M$CDN)	Sustaining Capital (M$CDN)	Total Capital (M$CDN)
Open Pit – Prestrip (capitalized)	94.7	-	94.7
Open Pit - Capital	8.4	6.7	15.2
Open Pit Mining Subtotal	**103.1**	**6.7**	**109.9**
Underground Mining		559.7	559.7
Processing	191.3	25.5	216.8
Infrastructure	42.1	22.2	64.3
Environmental		25.0	25.0
Indirects	64.4	8.2	72.6
Contingency	48.6	35.2	83.8
Total	**449.5**	**682.6**	**1,132.1**

1.13.2 Operating Costs

The life of mine operating cost summary is shown in Table 1-4.

Table 1-4: Troilus Gold Project Operating Cost Estimate ($CDN)

	Units	Open Pit Only (Year 1 – 5)	Open Pit & U/G (Year 1 – 14)	U/G Only (Year 15 – 22)	Life of Mine (Year 1-22)
Open Pit Mining	$CDN /t moved	2.73	2.70		2.70
	$CDN /t moved	15.62	12.62	-	12.62
Underground Mining	$CDN /t moved		19.54	19.26	19.38
Processing	$CDN /t moved	6.74	6.74	6.74	6.74
G&A	$CDN /t moved	1.48	1.60	4.19	1.92
Concentrate Trucking	$CDN /t moved	0.26	0.32	0.26	0.32
Total Operating Cost	**$CDN /t moved**	**24.10**	**22.05**	**30.45**	**23.08**

Diesel and electricity pricing was obtained locally and is $CDN 1.03/l and $CDN 33.00/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 $CDN 0.41/t moved life of mine or $CDN 1.91/t milled.

General and Administrative costs consider a camp operation with a local quotation.

1.14 Financial Analysis

A pre-tax and post-tax cash flow model was prepared by AGP on behalf of Troilus incorporating Quebec and Federal Tax rules. Input metal prices and the results are shown in Table 1-5.

The results indicate a post-tax NPV(5%) of $CDN 778M ($US 576M) with an IRR of 22.9% and payback period of 4 years. Initial capital is $CDN 449.5M ($US 333M) with life of mine capital totaling $CDN 1,132.1M ($US 838.6M).

Table 1-5: Troilus Gold Project – Discounted Cash Flow Financial Summary ($CDN)

Parameter	Units	Pre-Tax	Post-Tax
Metal Prices			
Gold	$US/oz	1,475.00	
Copper	$US/lb	3.00	
Silver	$US/oz	20.00	
Exchange Rate	$CDN:$US	0.74	
Net Present Value (5%)	$CDN M	$1,311	**$778**
Internal Rate of Return	%	29.6	**22.9**
Net Revenue less Royalties	$CDN M	8,322.4	**8,322.4**
Total Operating Cost	$CDN M	4,443.0	**4,443.0**
Life of Mine Capital Cost	$CDN M	1,132.1	**1,132.1**
Taxes	$CDN M	-	**1,038.8**
Net Cash Flow	$CDN M	2,747.3	**1,708.5**
Payback Period	Years	3.7	**4.0**
Cash Costs (with credits)	$CDN/oz	970	**1,241**
All-in Sustaining Cost	$CDN/oz	1,148	**1,419**
Payable Metals (Life of Mine)			
Gold	Moz	3.84	
Copper	M Lbs	265	
Silver	Moz	1.47	
Initial Capital	$CDN M	449.5	
Sustaining Capital	$CDN M	682.6	
Total Capital	$CDN M	1,132.1	
Mine Life	Years	21	

1.15 Recommendations

1.15.1 Geology

It is recommended that continued delineation drilling continue at the Z87 and J4/J5 Zones. Specifically, within the area between the two Zone, and at depth and long strike at the Z87 and J4/J5 Zones. Current interpretations indicate a continuity of mineralization between the Z87 and J4/J5 Zone and does not have sufficient drilling information to determine the geology.

Both the Z87 and J4/J4 Zones seem to show continued mineralization along strike and at depth that both infill and delineation drilling would support the current interpretation and possibly show an increase the mineral resources. Approximately 45,000 m of drilling is proposed for these zones: between 85-95 drill holes.

It is recommended further drilling continue at the SW Zone. The deposit seems to show continuity of mineralization along strike of both limbs of the interpreted synclinal fold. Both infill and delineation drilling is expected to upgrade the resources to an Indicated category. Approximately 16,000 m of drilling is proposed: between 55-60 drill holes.

The estimated budget for these proposed exploration programs would be approximately $CDN 13.6 million.

1.15.2 Geotechnical

The following items are recommended to advance the geotechnical information to the level of Prefeasibility:

1. Geotechnical Drilling, Laboratory and Fieldwork, including:
   - Drilling – 5,050 metres of drilling at $CDN 200/m = $CDN 1,010,000
   - Geotechnical Logging - $CDN 126,250
   - Downhole Testing - $CDN 100,000
   - Laboratory Testwork - $CDN 25,000
   - Verify and validate current geologic features
2. Seismic Study
   - Determine the seismic loading and apply to updated geologic structures to determine stability concerns if any
3. Slot and Mass Blast Analysis
   - Evaluate performance of hangingwall rock and S+MB mining method in detailed analysis of drawdown of stopes
   - Provide guidance on drawdown rates from Slot and Mass Blast stopes which affects production rate of underground mining
   - Characterize the rock mass as part of that analysis
4. Hydrogeological Analysis
   - Collect and interpret the data
   - Provide mine engineering guidance for dewatering systems.

Items 2 thru 4 are estimated at ~$CDN 125,000. The majority of this analysis and study is included in the Prefeasibility study cost estimate described later, except for the drilling program (Item 1). That cost of $CDN 1,261,250 is above this and should be included in the budget separately.

1.15.3 Mining

Open Pit

The open pit design work benefited from the experience of the previous operation. In particular, the knowledge gained on pit slopes that exist to this day. As well, the current status of the waste dumps and their stability reaffirm the design criteria. Building on that knowledge and the work completed in the PEA, the following is recommended for advancing the open pit design work to a Prefeasibility level of study:

1. Blasting Study – to fine tune fragmentation

2. Equipment Costs and Fleet Selection – examine alternate equipment to reduce costs
3. Ore Sampling Protocols – determine proper sampling for grade control
4. Pit Electrification Optimization – placement of electrical lines for maximum benefit

These recommendations are typically included in the normal cost of open pit design and engineering; therefore, no additional budget is listed beyond that which is allocated for the Prefeasibility study.

Underground

The current design in the PEA for the underground mining portion considers the use of sublevel caving (SLC) and slot and mass blast (S&MB) stopes. Additional detailed work will be required for the areas using these methods. The following is recommended to bring the level of study up to Prefeasibility:

1. Open Pit and Underground Interface Study – optimize grade extraction at interface
2. Drilling and Blasting Study – determine proper fragmentation in SLC and S&MB stopes
3. Rail-Veyor Detailed Studies – examine use earlier to minimize development costs
4. Dewatering Study – detailed water handling study
5. Contract Mining – examine contract mining cost effectiveness
6. Labour Study – complete salary and manpower availability survey

Many of the recommendations for the underground mine design would be covered under the Prefeasibility engineering study budget mentioned later. The labour study, as it is used by various disciplines is highlighted here as a cost above what is included in that estimate. The cost is shown here for the overall budget estimate. That cost is estimated at $CDN 100,000.

1.15.4 Metallurgy

Additional testwork should be completed, with a focus on samples from each deposit included within the PEA mine plan, and with a focus on lower grade sample characterisation. This work would for the most part be conducted at the laboratory scale, on representative samples of drill core. Flotation work may require larger scale primary flotation testing, in order to generate sufficient rougher concentrate for adequate cleaner flotation characterisation.

The extent of the deposit would also suggest the adoption of a geometallurgical approach to the metallurgical characterisation.

Testwork should include the following:

- additional comminution testwork, including crusher work index confirmation (gyratory crusher sizing). A larger database of SMC results, for example, would assist in determining variability within and/or between the deposits
- determination of modal mineralogy plus gold deportment
- gravity testwork, including Extended GRG (E-GRG) characterisation
- flotation testwork, using larger (10-kg) test charges to ensure sufficient metal units in locked cycle cleaner circuit evaluations; gravity concentration of rougher concentrates would be an option, although concentrate mass requirements may limit the extent of this work. Flotation testing should allow for concentrate copper grade vs copper and gold

recovery target optimized concentrate copper grades, as determined through discussion with potential smelters

- determination of minor (deleterious) element concentrations in flotation concentrates
- additional cyanidation testing, plus flotation testing on cyanidation residues
- cyanide detoxification testing
- environmental characterisation work on tailing products. This would include ABA characterisation, metals leaching work, humidity cell
- physical characterisation of tailing products

The metallurgical work detailed above is expected to cost $CDN 500,000.

1.15.5 Infrastructure

With the addition of several new or realigned infrastructure items over the past operation, further study will be required. These additional studies should include the following:

- No Name Creek dyke by the SW Pit
- No Name Creek diversion ditch realignment
- Tailings facility
- High Voltage Line pole realignment near tailings
- New access road alignment
- Detailed surveys of plant site, crusher, diversion ditch and new waste dump foundations

This work will also include incorporation of the previously discussed geotechnical work. These studies and surveys are estimated to cost $CDN 500,000.

1.15.6 Environmental

Troilus Gold has an advanced understanding of the environmental concerns at the project site from the past operations and ongoing monitoring. This level of information is currently beyond what is normally associated with a PEA study and well advanced for a Prefeasibility study.

Additional background information needs to be collected, especially regarding the creek diversion realignment, future dyke by the SW pit, the SW pit area and expansion of the tailings and potential discharge. Further study will assist in providing regulators with the required additional information necessary for permitting of the proposed project.

This additional study work will require outside support beyond the current Troilus Gold teams work. An estimate of this work is $CDN 300,000 to prepare for the Prefeasibility study.

1.15.7 Estimated Budget

The level of resource classification and historical information available at the Troilus Gold project is beneficial in reducing the cost of further studies as only updates are required in some disciplines. Completing this work and combining the results of the various disciplines of geology, geotechnical, metallurgy, mining and environmental will be the focus of the Prefeasibility study lead. This work by all the disciplines beyond the previous mentioned studies is estimated to be in the order of $CDN 2-3 million.

The total estimated budget for the prefeasibility study and associated work is outlined in Table 1-6.

Table 1-6: Estimate of Recommended Budgets and Prefeasibility ($CDN)

Area of Study	Approximate Cost ($CDN)
Geology	$13,600,000
Geotechnical	$1,261,250
Underground Mining	$100,000
Metallurgy	$500,000
Infrastructure	$500,000
Environmental	$300,000
Prefeasibility Study	$3,000,000
TOTAL	**$19,261,250**

2 INTRODUCTION

Troilus is a Canadian exploration and development company, based in Toronto, Canada, and is publicly-listed on the Toronto Stock Exchange (TLG.TSE). Troilus is focused on the development of the Troilus Gold Project, which includes the historic Troilus Mine; and the Troilus Frotêt Project. Troilus holds a 100% interest in the mineral rights for the Projects. The Projects are located in central Quebec and are situated approximately 120 km north of Chibougamau.

The Troilus Mine was an open pit operation producing gold, copper, and silver continuously from November 1996 to April 2009. The Troilus Mine produced over two million ounces (oz) of gold and approximately 70,000 tonnes (t) of copper. After the mine ceased production in 2009, the 20,000 tonnes per day (tpd) mill processed low grade stockpiles until June 29, 2010. Following this, the mill was sold and shipped to Mexico and the main camp facilities were dismantled in late 2010.

2.1 Terms of Reference

This Technical Report and Preliminary Economic Assessment (PEA) was prepared by AGP for Troilus to present the results preliminary economic evaluation of Troilus Gold Project. This Technical Report was prepared for the Troilus Property in accordance with NI 43-101 and Form 43-101F1. The mineral resources used in the PEA were prepared on the Z87, J4/J5, and Southwest (SW) Zone within the Troilus Gold Project.

All units of measurement used in this technical report and resource estimate are in metric, unless otherwise stated. All grid references are based on the NAD83 Datum (NAD83) UTM coordinate system. All currency units are in Canadian dollars unless otherwise stated.

2.2 Qualified Persons

The list of Qualified Persons (QPs) responsible for the preparation of this technical report and the sections under their responsibility are provided in Table 2-1:

Table 2-1: Summary of QPs and Responsibilities

QPs	Position	Report Sections
Gordon Zurowski, P.Eng.	Principal Mining Engineer	1.1, 1.8 – 1.12 1.13.2, 1.14.2, 15, 16, 17, 18, 19, 20, 21 (except 21.2.3 and 21.3.3), 22, 24, 25.2, 25.3,25.5, 25.6, 26.1, 26.3, 26.4, 26.5, 26.7, 26.8, 26.9, 26.10
Mr. Paul Daigle, P.Geo.	Senior Associate Geologist	1.2 – 1.6, 1.15.1,4-12, 14, 23, 25.1, 26.2, 27
Mr. Andy Holloway, P.Eng.	Principal Processing Engineer	1.7, 1.9, 1.15.4,13, 17, 21.2.3,21.3.3, 25.4, 26.6

2.3 Site Visit

2.3.1 Geology

Mr. Daigle conducted a site visit to the Property from February 18 to February 20, 2020. The Project site was inspected for two days during the site visit. During the site visit, the 2019 – 2020 drill program was in progress on the SW Zone.

Drill core logging, sampling, and storage facilities were inspected during the site visit. The site visit also included verifying drill hole collar locations and a review of drill logs against selected drill core. Mr. Daigle was accompanied on site by:

- M. Bertrand Brassard, Chief Geologist for Troilus
- M. Thiago Diniz, Technical Manager for Troilus.

2.3.2 Mining

Mr. Zurowski conducted a site visit to the Property from July 13 to 15, 2020. The Project site was inspected for 2 days during the site visit.

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:

- M. Bertrand Brassard, Chief Geologist for Troilus
- Daniel Bergeron, Vice-President, Quebec Operations for Troilus
- Jacqueline Leroux, Director of Environment for Troilus

2.4 Effective Date

The report has multiple effective dates as noted below:

- The effective date of the mineral resources for the Troilus Project is 20 July 2020.
- The effective date of the PEA for the Troilus Gold project is August 31, 2020.

There were no material changes to the scientific and technical information on the project between the effective data and the signature date of the report.

2.5 Information Sources and References

The main sources of information in preparing this report are based on information located within internal reports obtained from Troilus. Information, conclusions, and recommendations contained herein are based on a field examination, including a study of relevant and available technical data, including, and not limited to the numerous reports listed in the Reference section. This report is prepared with the most recent information available at the time of study.

AGP validated the resource estimates for the Z87 Zone and J4/J5 Zone originally estimated by Roscoe Postle Associates Inc. (RPA). The resource estimates were described in a report authored by Luke Evans dated December 20, 2020, titled "Technical Report on the Troilus Gold-Copper Project Mineral Resource Estimate, Quebec, Canada" (RPA (2019b). Since the report was published, there has been no further work conducted on the Z87 Zone and J4/J5 Zone.

The review focused on the drill hole database validation, assay validation against the laboratory certificate, wireframe validation, and grade interpolation. The purpose of this review is to validate the current model and provide recommendations for improvement of the grade estimate and/or improve the model classification. AGP accepts the validity of the model, which was not re-interpolated; therefore, the text and tables for these two zones were extracted from the previous NI 43-101 report from RPA (2019b).

All mineral resources described herein have been reported within updated constraining shells.

2.6 Previous Technical Reports

The Troilus Mine and Troilus Gold Project has been the subject of several technical reports. The previous NI 43-101 technical reports are found in the References section and summarized in Table 2-2 below:

Table 2-2: 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, Quebec, Canada
RPA, 2016	Jun 30, 2016	Sulliden Mining Capital Inc.	Technical Report on the Troilus Gold-Copper Mine Mineral Resource Estimate, Quebec, Canada
RPA, 2017	Nov 20, 2017	Pitchblack Resources Ltd.	Technical Report on the Troilus Gold-Copper Mine Mineral Resource Estimate, Quebec, Canada
RPA, 2019a	Jan 1, 2019	Troilus Gold Corp.	Technical Report on the Troilus Gold-Copper Mine Mineral Resource Estimate, Quebec, Canada
RPA, 2019b	Dec 20, 2019	Troilus Gold Corp.	Technical Report on the Troilus Gold-Copper Mine Mineral Resource Estimate, Quebec, Canada
AGP, 2020	Aug. 27, 2020	Troilus Gold Corp.	Technical Report and Mineral Resource Estimate on the Troilus Gold-Copper Project, Quebec, Canada

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 Troilus 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 Quebec 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 26 July 2020 found here:

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

The QP's 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.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location and Description

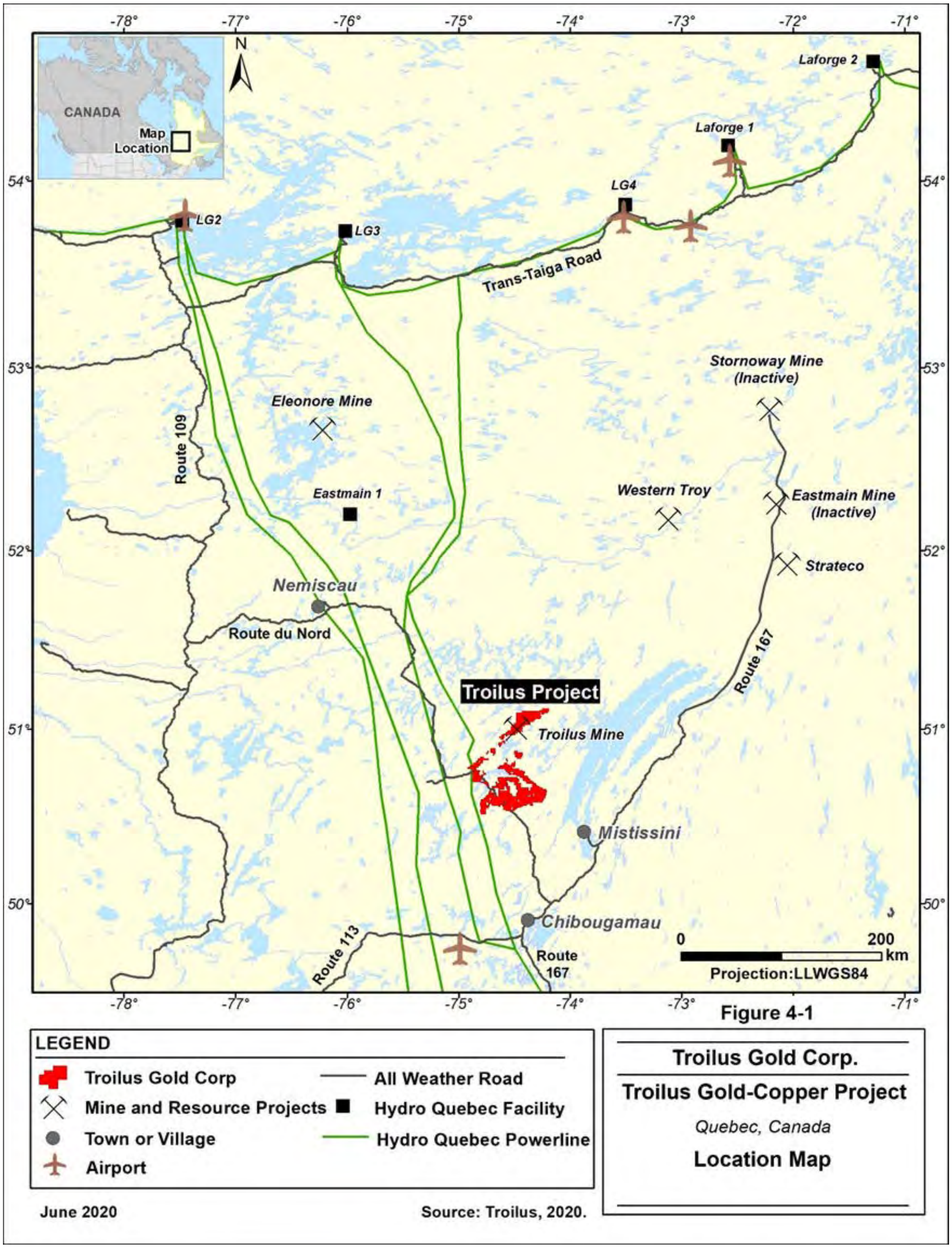
The Troilus Gold Property (Property) is defined by the mineral rights that are 100% held by Troilus. The mineral rights to the Property cover a total area of approximately 107,321 ha.

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 175 km north (by road) of Chibougamau
- in the Province of Quebec
- 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

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

Figure 4-1: Location Map, Central Quebec



Source: Troilus (2020)

The mineral rights for the Property are comprised of a single Mining Lease (*Bail Minier*), and 1,988 mineral claims (*Titres Miniers*). All mineral rights are in good standing.

The mineral rights for the Property are summarized in Table 4-1 below. Figure 4-2 presents a map showing all mineral rights held by Troilus. Figure 4-3 illustrates the mineral rights for the Troilus Gold Project.

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	11 Mar 2026	835.46
Mineral Claims (Troilus Gold Project)	2422145 – 2422147	3	Feb 2022	162.38
	2424713 – 2425732, 2424748 – 2424786, 2424958 – 2425037, 2488059	20 39 80 1	Mar 2022	7576.32
	1133905 – 1134008, 1133913 – 1133926, 1133929 – 1133930, 1133936 – 1133980, 1133982 – 1133985, 1133998 – 1134008, 2488138, 2488294 – 2488297	5 14 2 45 4 12 1 4	Apr 2022	4149.27
	2491523 – 2491527	5	May 2022	270.67
	2499212 – 2499223, 2500001 – 2500004	12 4	Aug 2022	865.28
	2502354 – 2502365	12	Sep 2022	648.78
	2504200 – 2504230	31	Oct 2022	1677.04
Mineral Claims (Troilus Frotêt Project)		1,695	Apr 2021 – Jun 2023	91,135.78
	Total	**1,989**		**107,320.98**

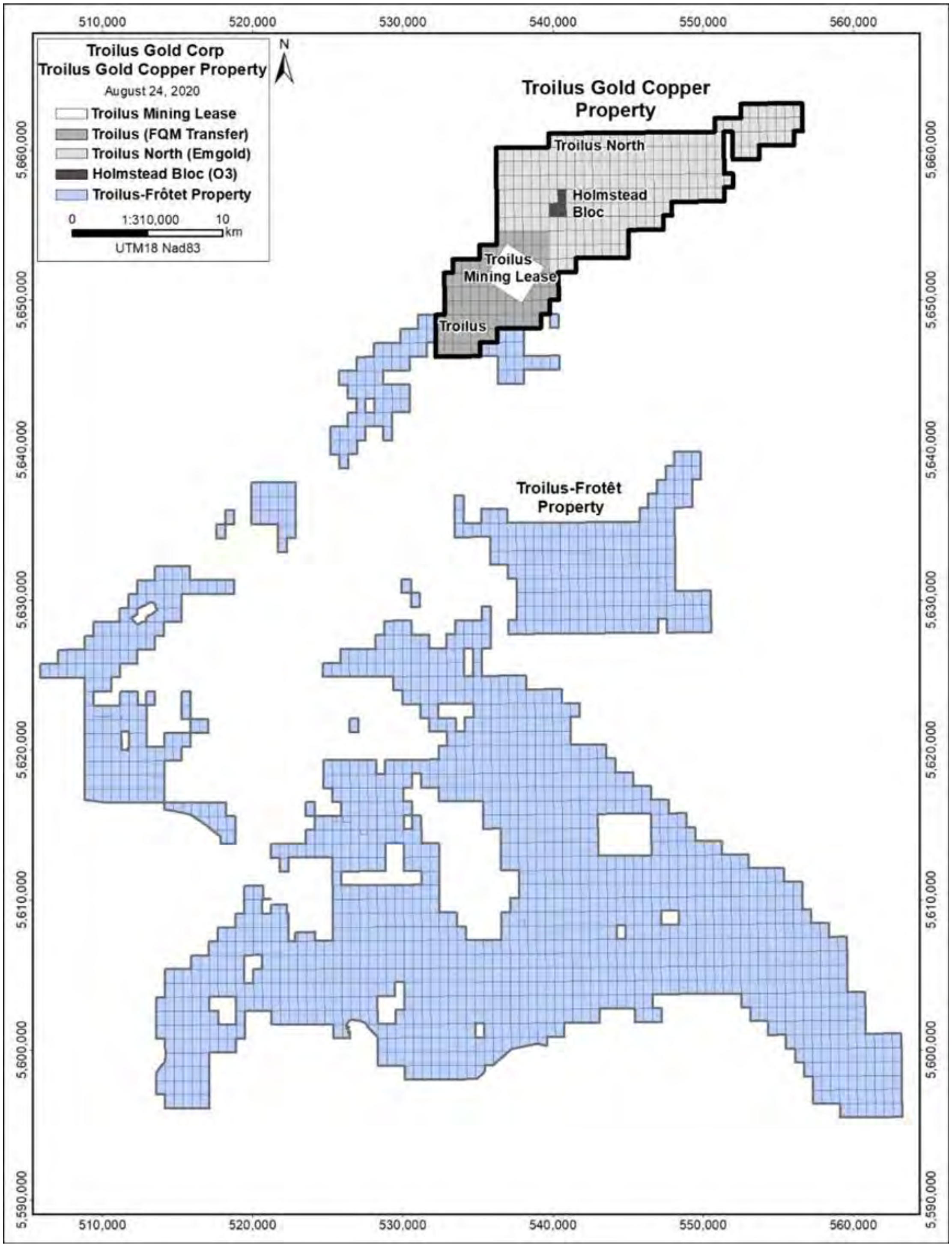
*list shows groupings of sequential mineral claim numbers

The Property is divided into two projects: The Troilus Gold Project and the Troilus Frotêt Project.

The Troilus Gold Project was acquired through three transactions. The first consisted of the acquisition of the one mining lease and 81 mineral claims, which collectively covered approximately 4,714 ha and included the former Troilus Mine from First Quantum Minerals Ltd. (First Quantum) in April 2018. The second transaction consisted of the acquisition of 209 mineral claims in the north half of the Property, covering approximately 11,309 ha from Emgold Mining Corp. (Emgold) in December 2018, whereby Troilus acquired the Troilus North property located immediately to the north and east of the Troilus property. The next transaction consisted of the acquisition of three mining claims, covering approximately 162 ha from O3 Mining Inc. (O3 Mining) in November 2019. These claims are labelled the Holmstead Bloc (Figure 4-3).

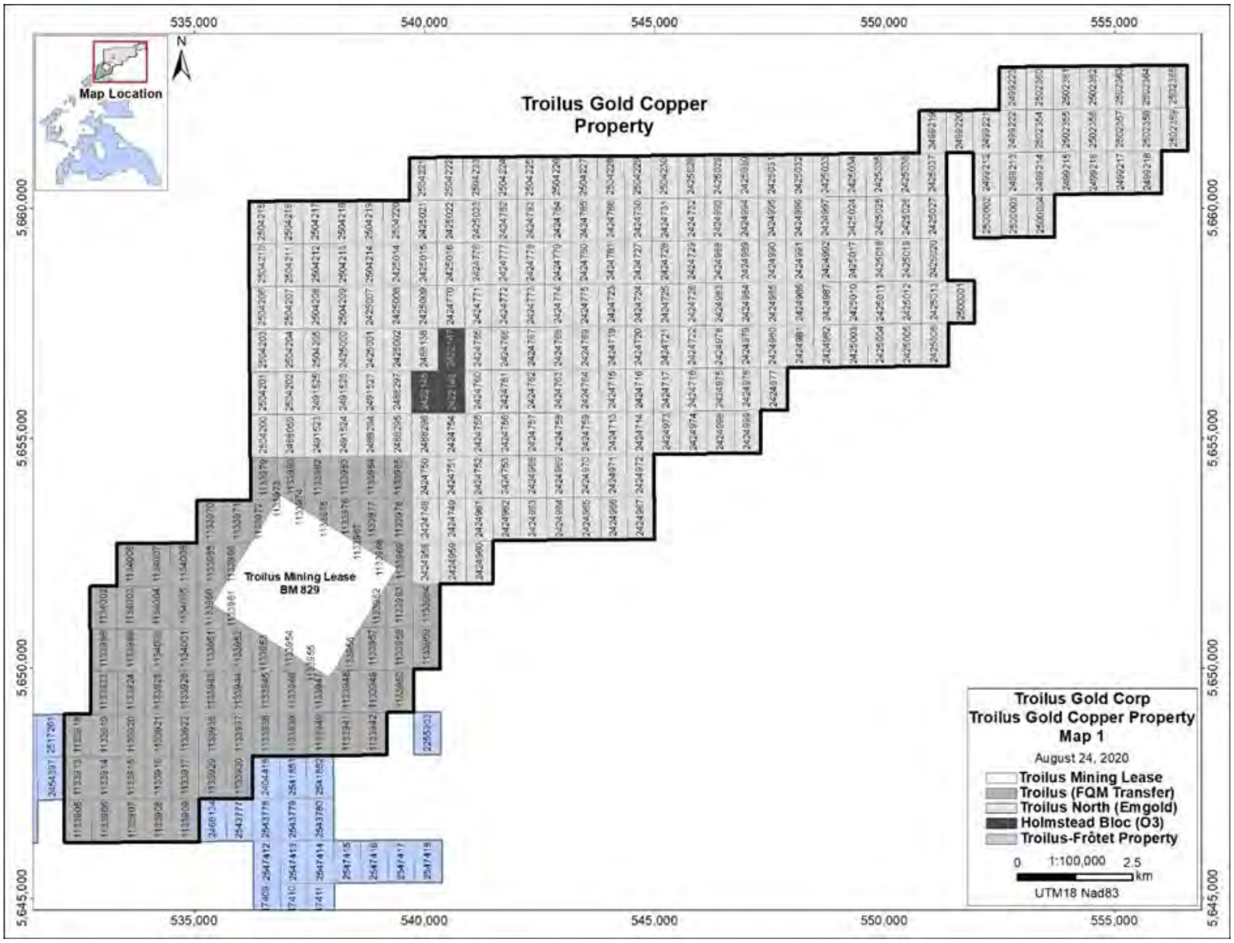
The Troilus Frotêt Project consists of 1,695 mineral claims southwest and south of the Property that covers approximately 91,136 ha. In April 2020, Troilus acquired an additional 627 claims from O3 representing 33,410 hectares. In July 2020, the Company acquired 91 mineral claims covering an area of approximately 4,960 ha from Globex Mining Enterprises: and 21 mineral claims covering an area of approximately 1,140 ha from 9219-8845 Qc. Inc. doing business as (dba) Canadian Mining House. All minerals rights are in good standing. Additionally, Troilus has staked a total of 956 mineral claims covering an area of 51,626 ha.

Figure 4-2: Mineral Rights Map; held by Troilus



Source: Troilus (2020)

Figure 4-3: Mineral Rights Map for the Troilus Gold Project



Source: Troilus (2020)

4.2 Project Ownership

On May 2, 2016, a wholly-owned subsidiary of Sulliden Mining Capital Inc., 2507868 Ontario Inc. (Sulliden Sub) entered into the Agreement with 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 Troilus property from First Quantum and title was transferred to Troilus. The 81 claims previously owned by First Quantum are 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. In addition, Nomad Royalty Company. has an additional 1% royalty, acquired from an arm's length private company in October 2019.

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 $250,000 in cash. The shares were subject to a four-month statutory hold period. Until December 5, 2020, Troilus has a Right of First Refusal (ROFR)whereby Troilus has the opportunity to find a buyer at equal or superior terms in the event Emgold wishes to dispose of the shares. During the ROFR period, provided Emgold holds no less than 5% of Troilus' issued and outstanding shares, Emgold shall have the right to participate in transactions involving the issuance of equity securities of Troilus, in order to maintain its proportional interest in Troilus, subject to certain conditions.

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

- a 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 has 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. Troilus has the right to repurchase a 1% NSR on the O3 Mining Inc. claims at any time for CAD$1,000,000. In addition, the O3 Mining Inc. claims are subject to a 2% NSR granted to Inco Limited (now Vale) on seven of the 627 claims and a 1% NSR granted to Falconbridge (now Glencore) on 73 claims comprising the Beaufield Property.

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 CAD$500,000 and to purchase the remaining 0.5% NSR on at any time for CAD$1,500,000.

4.3 Quebec Mineral Tenure

In Quebec, 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: Quebec Mining Act).

In Quebec, 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 previously owned by First Quantum are 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. In addition, Nomad Royalty Company has an additional 1% royalty, acquired from an arm's length private company in July 2020.

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

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

The three (3) mineral claims acquired from O3 Mining Inc. in November 2019, the Holmstead Bloc, are subject to the following royalties:

- a 2% NSR to O3 Mining Inc. that Troilus has the right to repurchase 1% of the NSR at any time for $1,000,000
- a 2% NSR to an individual, that Troilus has the right to repurchase 1% of the NSR at any time for $1,000,000

The 627 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
- 2% NSR granted to Inco Limited (now Vale) on seven of the 627 claims
- 1% NSR granted to Falconbridge (now Glencore) on 73 claims comprising the Beaufield Property

The 21 claims acquired from Canadian Mining House in July 2020 are subject a 1% NSR to CMH, 0.5% of which can be purchased by Troilus for $500,000 and 0.5% of which can be purchased by Troilus for $1,500,000.

The 91 claims acquired from Globex in July 2020 are subject to a 2% GMR (Gross Metal Royalty) to Globex, 1% of which can be purchased by Troilus at any time for $1,000,000.

4.5.2 Mine Restoration Plan

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 Quebec Ministry of Sustainable Development, Environment and Parks (Certificate of Authorization No. 3214-14-025) pursuant to modifications made November 3, 2010 and May 23, 2012.

Surface and groundwater water samples are taken at regular intervals at a number of 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 and J4 Open Pits and Waste Dumps; looking northwest

Source: Troilus (2018)

Figure 4-5: Troilus Z87 Open Pit; looking south

Source: Troilus (2018)

Figure 4-6: Troilus J4 Open Pit; looking north

Source: Troilus (2018)

Figure 4-7: Troilus J4 Open Pit; looking northwest

Source: Troilus (2019)

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 175 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 regional of the Property is situated in 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 mine. There are limited services available at Mistissini. In June 2018, Troilus opened an office at Mistissini. The provides a forum for exchanging information and liaising with the Cree on a variety of social, environmental, and economic aspects of the Project, in addition to the potential for future training, employment, and business opportunities. In October 2018, Troilus opened an exploration office in Chibougamau.

Chibougamau, population approximately 7,500 (est. 2016) is the largest town in Nord-du-Quebec, 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.

Politically, the province is very supportive of mining. The Quebec 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:

- a 50-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
- a number of tailings water pump houses
- gatehouse and gate

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 extent of the surface rights was sufficient to operate the mine in the past, however, additional surface rights may be as mineral resources are added to the current Project.

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 m above sea level (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 (>10 m) of fluvio-glacial till. Outcrops are sparse, and very large boulders sitting on surface are common.

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 Frôtet-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 Quebec. Some exploration work was conducted following this survey; however, no important discoveries were made.

6.1 Exploration and Development, Troilus Mine, 1985 -2010

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

Table 6-1: Summary of History of the Troilus Mine . 1985 - 2010

Date	Description
1985	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 options 50% interest from Kerr Addison and becomes operator.
December 1991	Kilborn Inc. Pre-Feasibility Study is negative (7,500 tpd).
February to May 1993	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 Mining Corporation through the public sale of its shares.
Late 1994	Construction commenced.
May 4, 1995	Metall changed its name to 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 tpd.
April 1998	Met-Chem 15,000 tpd mill expansion Feasibility accepted.
1999	Mill achieves 15,000 tpd.
2002	Mill achieves 16,000 tpd.
2004	Met-Chem 20,000 tpd mill expansion Feasibility accepted.
2005	Mill achieves 20,000 tpd.
2007	Underground ramp stopped at 519.1 m from portal on January 22, 2007.
2008	Mining at J4 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 April 13, 2009.
2010	Mill stopped on June 29, 2010.
2010	Mill sold and shipped to Mexico in September 2010.
2010	Camp sold on November 19, 2010 and subsequently dismantled.

6.1.1 Ownership History

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

In February 1993, Metall Mining Corporation (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 Mining Corp. (Inmet).

Inmet was acquired by First Quantum in March 2013. On April 8, 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 Corp. and Minnova, 1985 – 1993

In 1985, Kerr Addison acquired a large block of claims following a geological mapping program by the Quebec 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 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), and a number of 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 tonnes averaging 2.3 g/t Au was taken from the centre of Z87 and approximately 100 tonnes were treated at the pilot plant of the Centre de Recherche Minérale du Québec in Quebec City as part of a pre-feasibility study. The remaining 100 tonnes 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 Mining Corp, Inmet Mining Corp, 1993 – 2005

In February 1993, Metall Mining Corp. (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 tpd 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 Mining Corp. (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 tpd.

In April 1998, Inmet approved a 15,000 tpd mill expansion feasibility study by Met-Chem Canada Inc. (Met-Chem). Modifications to the mill started in December 1998, and the full 15,000 tpd 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 tpd. Modifications to the mill were completed in December 2004 and the full 20,000 tpd 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 tpd with a flow sheet consisting of a gravimetric 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 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.

The overall mill recovery averaged 83% for gold and 89% for copper. The Troilus Mine produced over two million ounces of gold and almost 70,000 tonnes 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

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	**18,389**
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	**7,640**
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	**69,639**
Total Excavated (000 t)	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	**230,368**
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	**1.00**
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	**0.10**
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 (ozs)*		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	

Note: Recovered metal after milling and smelter and refining adjustments

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

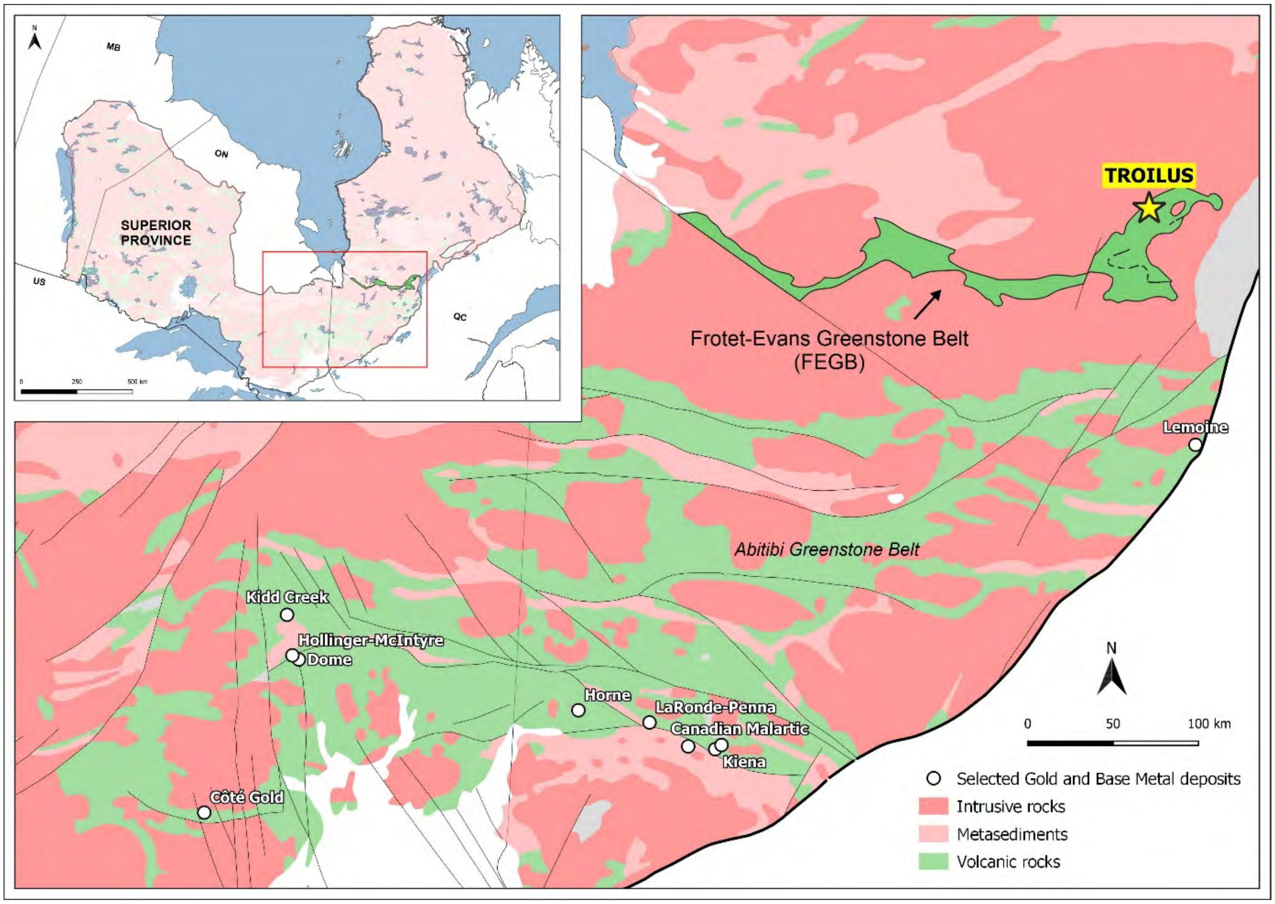
The Troilus Gold deposit lies within the eastern segment of the Frotêt-Evans Greenstone Belt (FEGB), in the Opatica Subprovince of the Superior Province in Quebec (Figure 7-1).

The Frotêt-Evans greenstone belt 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 Quebec. Boily and Dion (2002) divided the FEGB in four distinctive segments: (1) Evans-Ouagama, (2) Storm-Evans, (3) Assinica, and (4) Frotêt-Troilus. The eastern domain is known as Frotêt-Troilus (Simard, 1987) and has received most of the attention due to its larger economic potential (Figure 7-2).

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.

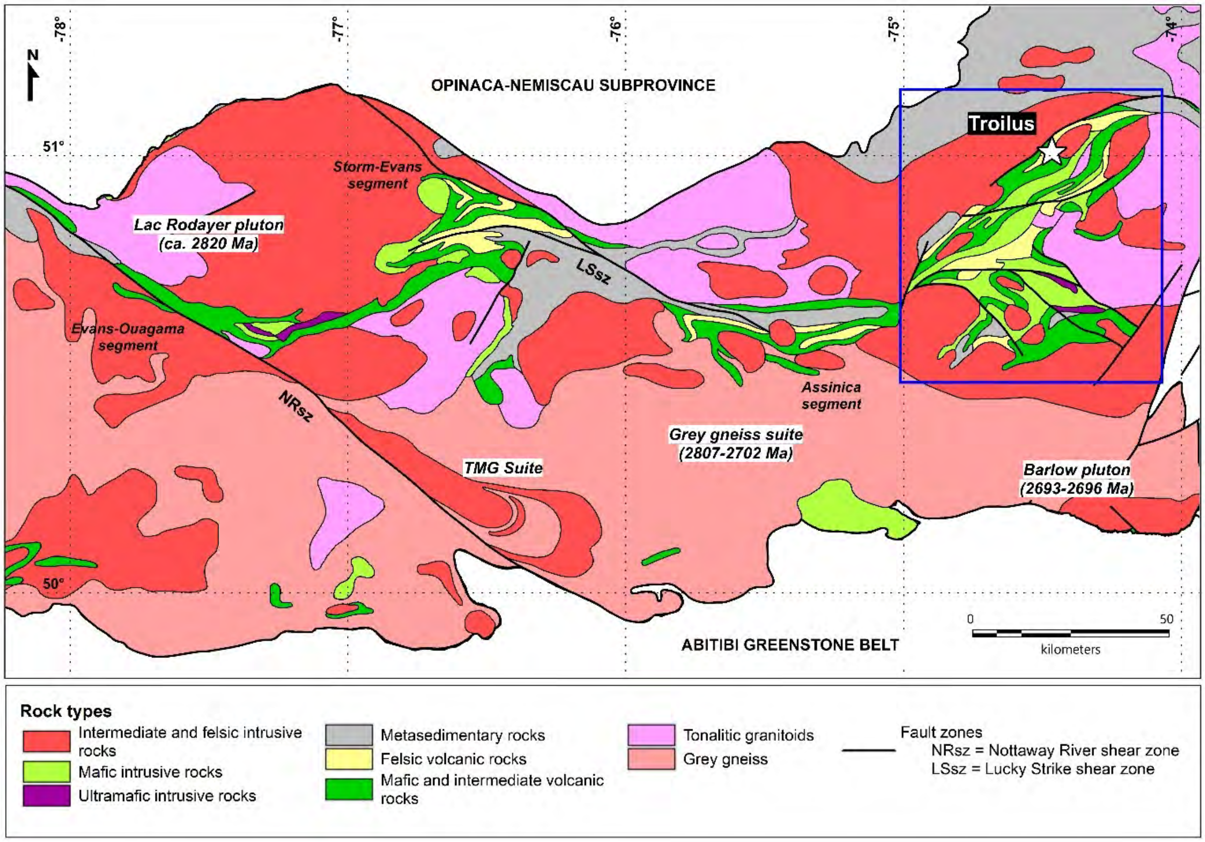
The few published U-Pb dates in zircon constrained the age of the FEGB between 2793 Ma and 2755 Ma (Pilote et al., 1997 in Boily and Dion, 2002). The circa 2793 Ma age is coincident with the dates obtained for the Troilus diorite.

Figure 7-1: Regional Geology Map



Source: Troilus (2019)

Figure 7-2: Regional Geology Map; Central Quebec



Source: Troilus (2019)

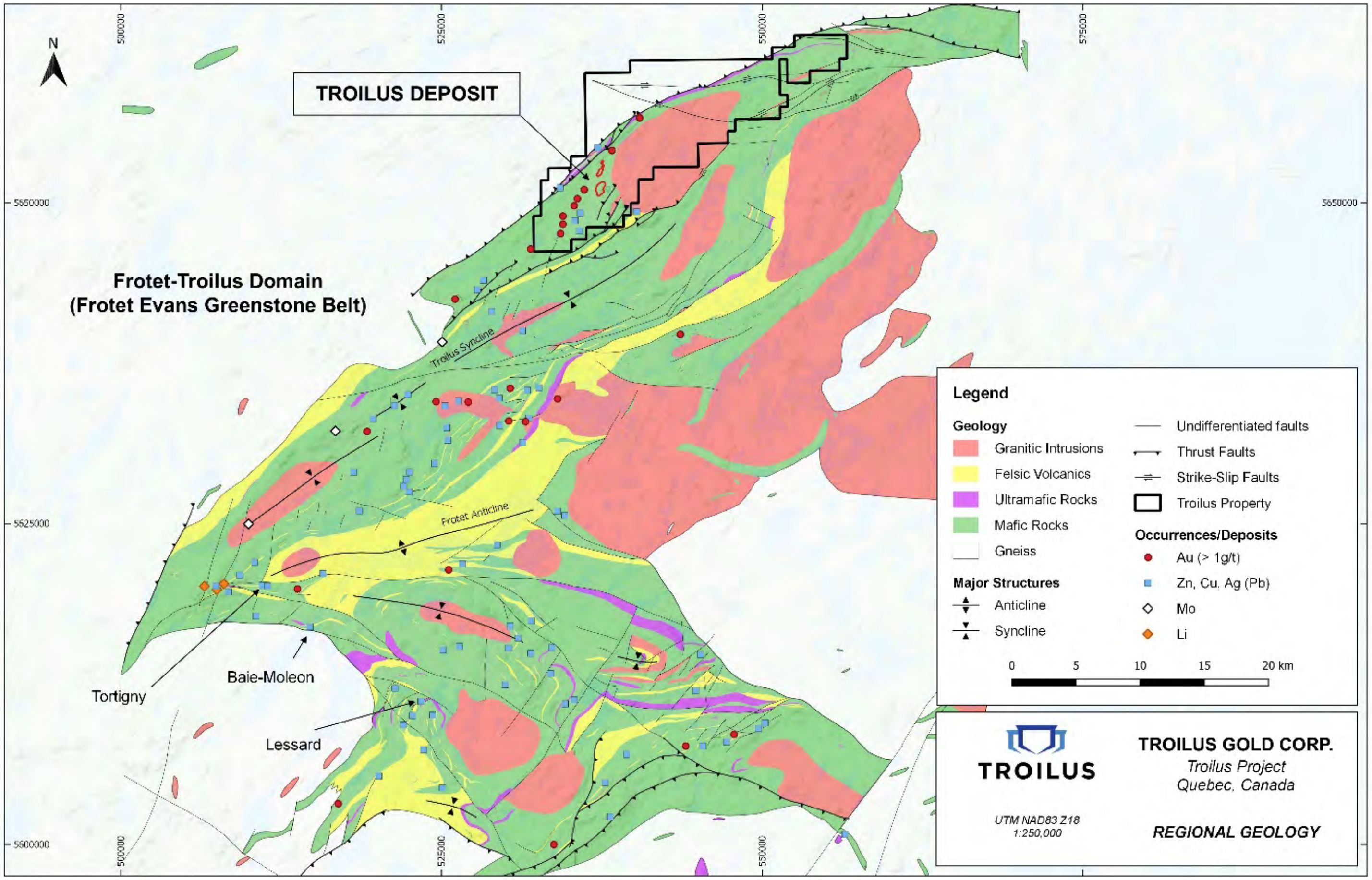
The Frotêt-Troilus domain (Figure 7-3) comprises the east domain of the FEGB and hosts the Troilus deposit. It is characterized by a complex and variable volcano-magmatic history, dominated by mafic volcanic rocks and coeval, cogenetic mafic intrusions, intermediate to felsic volcanic rocks and associated 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, that 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; Frotêt-Evans Greenstone Belt



Source: Troilus (2020)

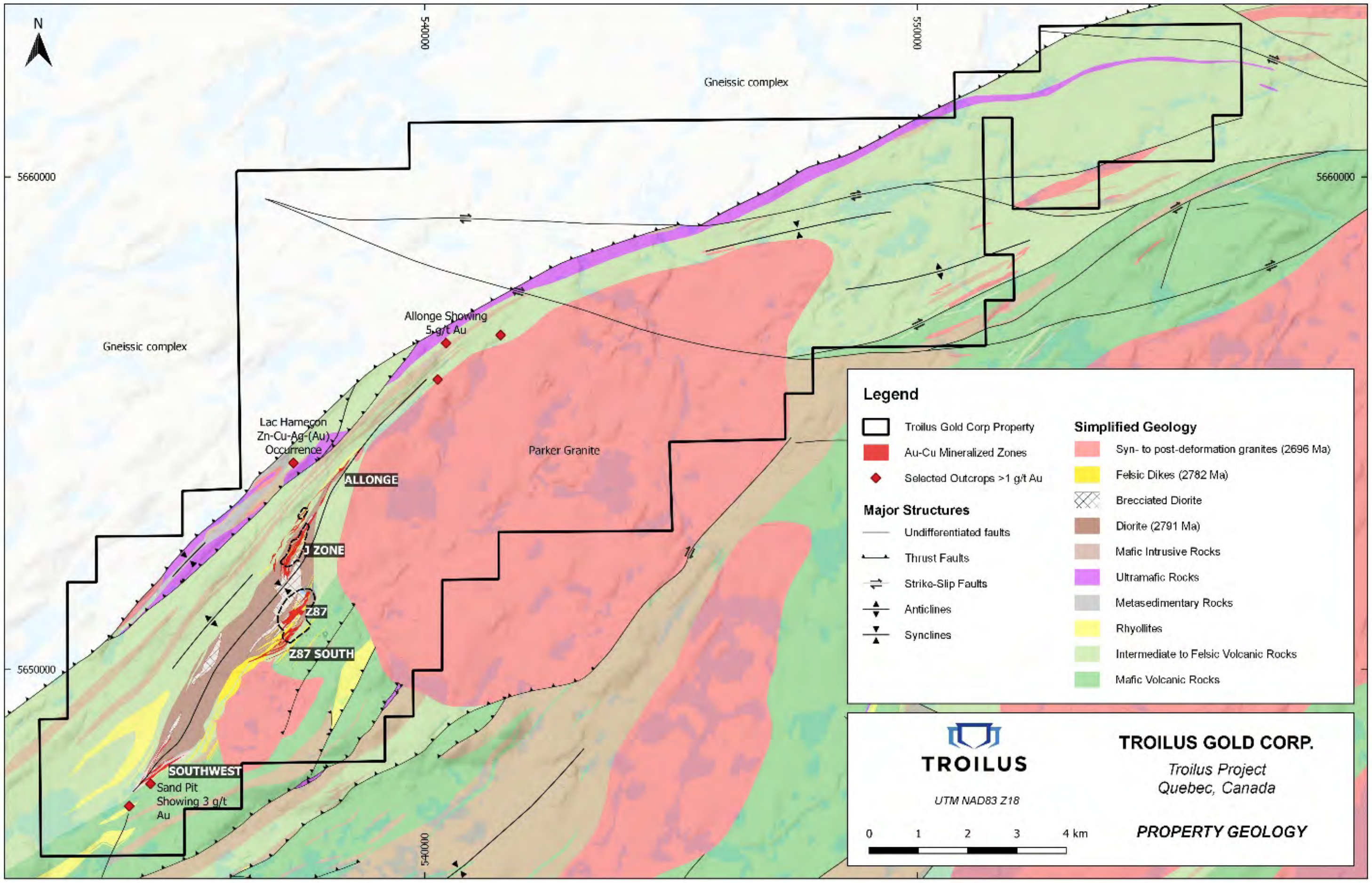
7.2 Project Geology

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- southwest striking thrust fault zones, parallel to the main regional foliation and to the volcanic bedding.

On the property (Figure 7-4), the Troilus Group is represented by a thick volcanic sequence, predominantly mafic to intermediate in composition, with local felsic flows and tuffs. Synvolcanic magmatism is marked by a series of gabbro and ultramafic sills. The main lithotypes which comprise the Troilus deposit region are a metadioritic pluton, an amphibolite, and a brecciated unit, which are all crosscut by a series of felsic dikes (Figure 7-5). Late-stage dikes 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.

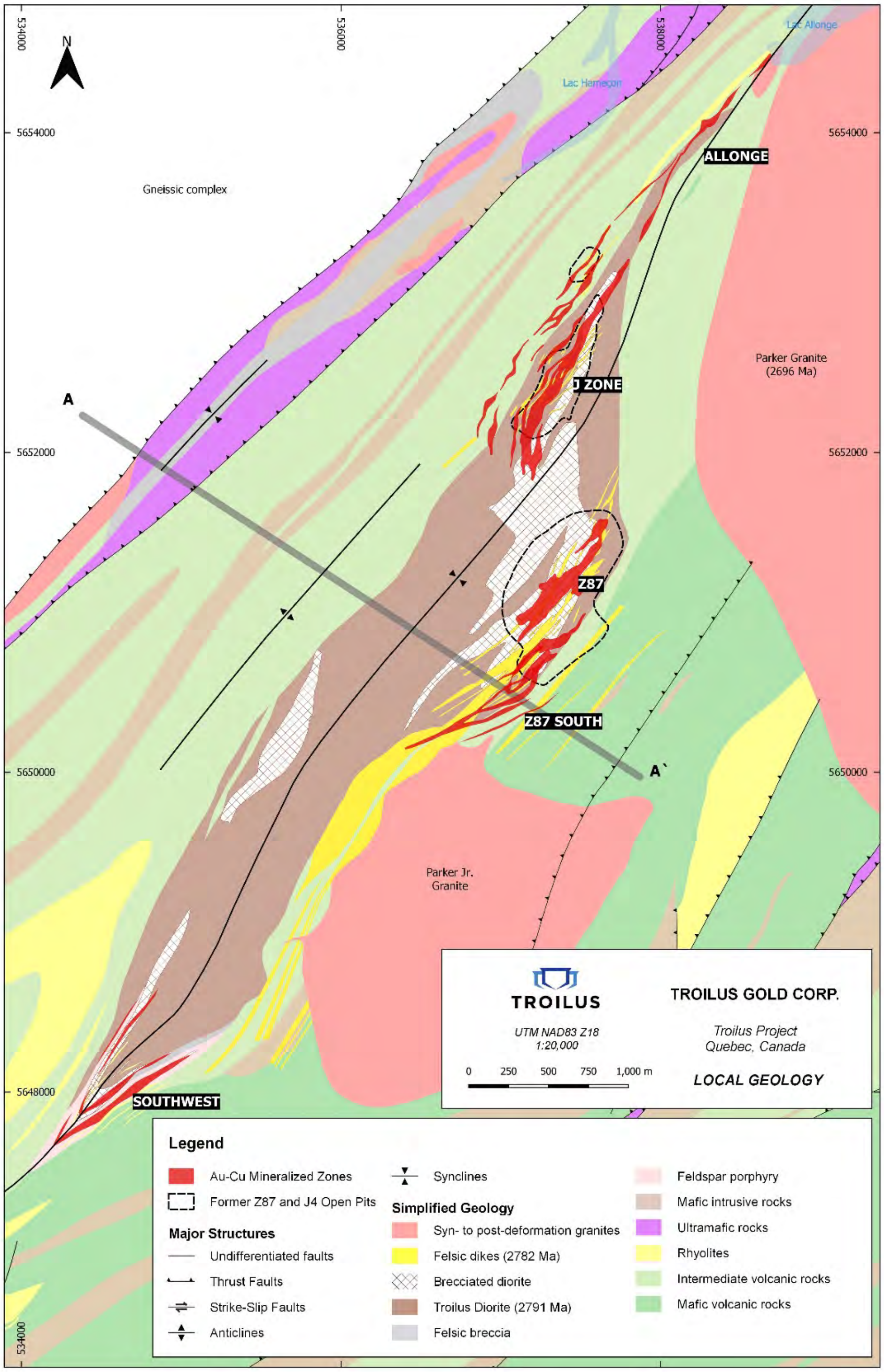
The following descriptions for the main lithologies, alteration, mineralization, and structural features are based mostly on a recent description of the 2018 and 2019 drill holes observations by the Troilus Gold geology team, as well as contributions from the works of Brassard (2018), Brassard & Hylands (2019), Diniz (2019), Laurentia Exploration (2018), and SRK (2018).

Figure 7-4: Geology Map; Troilus Gold Project



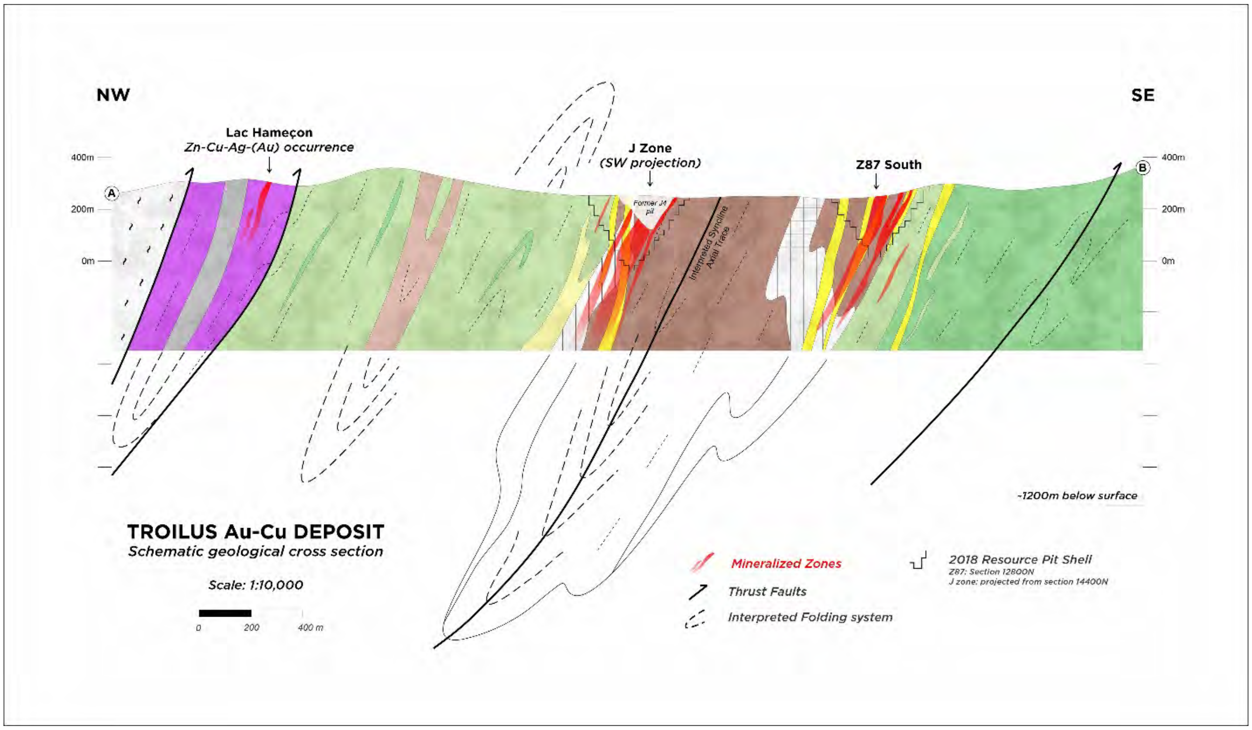
Source: Troilus (2020)

Figure 7-5: Geology Map; Z87, J4/J5 Zone and SW Zone



Source: Troilus (2020)

Figure 7-6: Project Geology Map; Schematic Cross-section



Source: Troilus (2020)

7.2.1 Local Lithological Units

Four main lithological units are recognized in the Troilus deposit region, broadly divided in: (i) mafic to felsic volcanic sequence; (ii) diorite and brecciated diorite; (iii) cross-cutting felsic dikes, and (iv) mafic to ultramafic intrusive. A series of distinct younger, post-deformation granitic intrusions crosscut all other lithotypes.

Mafic to Felsic Volcanic Sequence

Dominantly occurring throughout the entire Troilus property, and surrounding the Troilus deposit region, is a thick sequence of volcanic rocks of variable composition. The south-eastern 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. Locally, and especially observed in drill cores, the mafic volcanic rocks often display a compositional millimetric to centimetric banding, marked by alternating amphibole-rich green- to dark-green layers, with light-green or white-greyish feldspar and epidote-rich bands (Figure 7-7, photo A). In the deposit region, this lithotype is recognized on the footwall zones of Z87 and Z87S.

The basalt sequence is overlain, in gradual contact, with a more intermediate to felsic composition banded and laminated sequence, as it can be observed in drill cores of Z87S (Figure 7-7, photo E). In this sequence, quartz-feldspar-rich bands and layers are dominant over light-green amphibole layers.

Local garnet-rich quartz-rich intervals resembling volcaniclastic rocks occur towards the top of the sequence, as well as amorphous quartz-bands that could represent exhalative horizons.

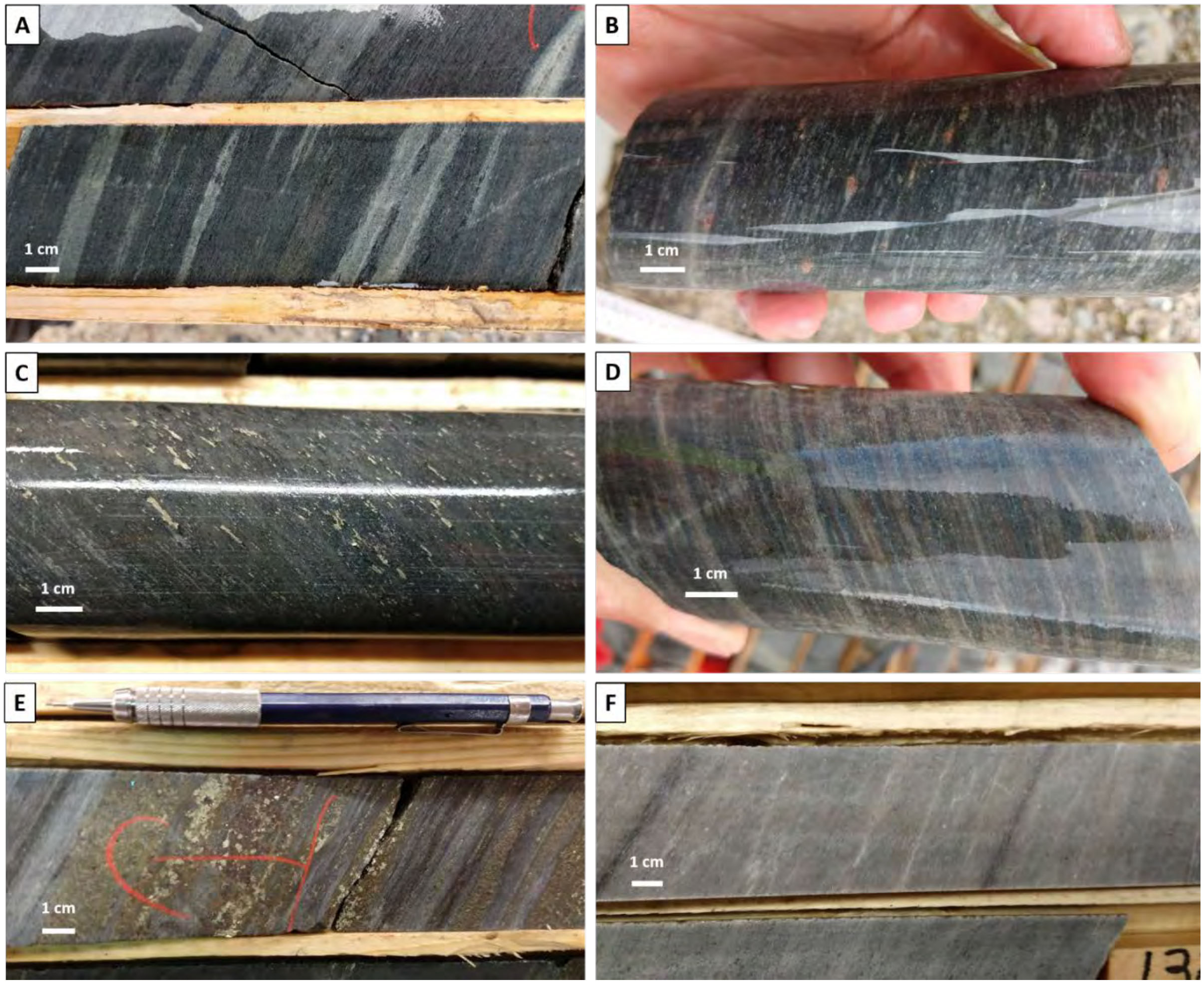
In the hanging wall portion of the J zones, the volcanic sequence is mainly represented by a finely laminated intermediate rock, grey to light-green in colour, often showing quartz and pink-garnet-rich horizons, that probably represent more volcaniclastic units of the sequence (Figure 7-7, photos B, C, and D). In the southern portion of the J4 pit, an amphibole-rich, volcaniclastic brecciated unit is present, containing intensely altered, irregularly shaped epidote-feldspar-rich clasts. The matrix is locally rich in magnetite.

Metric to decametric-scale lenses of rhyolite are identified within the volcanic sequence, and mainly occur bordering the diorite intrusion in its western margin. White, massive rhyolites outcrop in the southwestern region of the deposit, in the Southwest and Z86 zones, and have also been described in the hanging wall of the J4 pit (Figure 7-7F). They often display an intense sericite alteration, and typically contain millimetric quartz-filled vugs, surrounded by an aphanitic quartz-feldspar matrix.

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. Previous geologists have described an amphibolitic unit immediately surrounding the diorite

intrusion, part of which could represent a metamorphic equivalent of mafic volcanic rocks. A foliated amphibole-rich rock with a penetrative schistosity has also been described in boreholes in the footwall of Z87 and amphibolite is observed in the footwall of Z86 South (Z86S) in the Sand Pit.

Figure 7-7: Drill Core Photographs; Showing Mafic to Felsic 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. Felsic volcanics, sulfide rich (Py-Po-Sph), Z87 South
F. Rhyolite, J4

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, entirely surrounded by the volcanic sequence. It represents the main host rock for the mineralization at the Z87, Z87S and J zones. It comprises a pale to greenish-grey rock, composed predominantly of medium to coarse grained crystals of plagioclase and hornblende dispersed in a fine-grained groundmass of feldspar, amphibole, epidote, and quartz (Carles, 2000).

The Z87 hanging wall is mainly represented by brecciated diorite. Metre-scale intervals of massive, coarse to fine grained diorite, as well as porphyritic diorite, alternate with the typical brecciated diorite. The breccia is unsorted and predominantly matrix-supported (Figure 7-8), being characterized by two types of centimeter to decimeter scale pale coloured fragments: (i) massive diorite; and (ii) porphyritic diorite. Overall, 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. 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. A transition from massive to fractured to brecciated diorite has been locally observed in drill core, as well as in boulders around the former open pits.

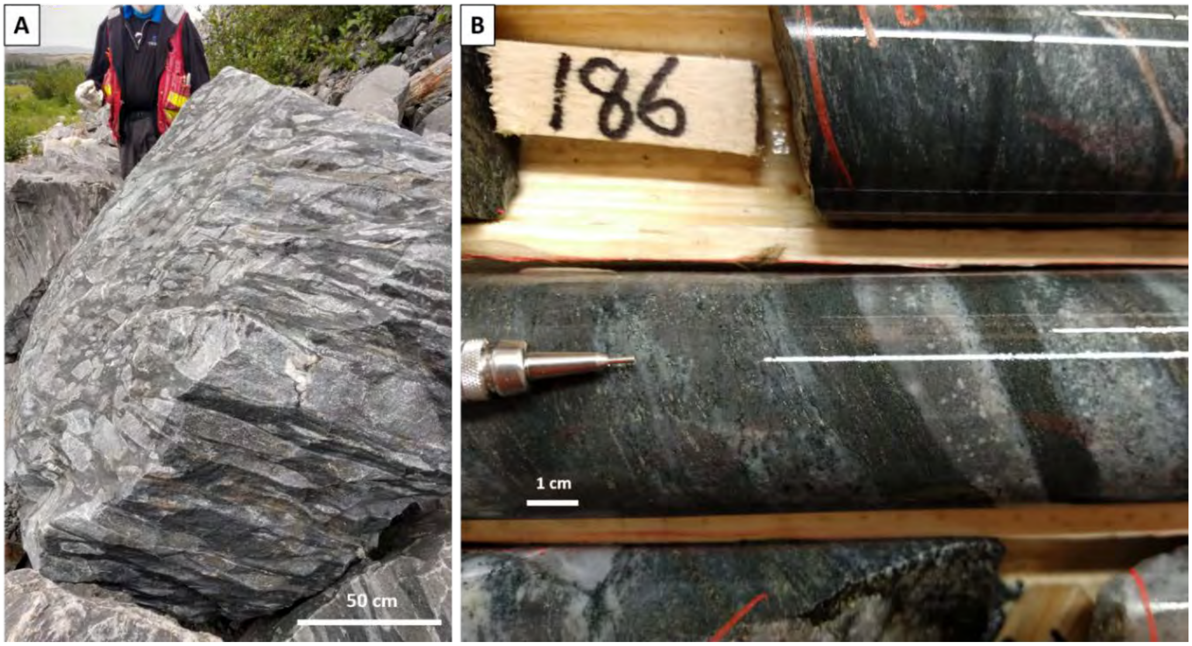
In the J zones, the diorite is predominantly fine grained, and biotite-rich, particularly within the mineralized intervals. Local metric to decametric-scale, coarse grained to porphyritic diorite are observed in drill core, particularly in the hanging wall of the mineralization. Deep drill holes in the southern portion of J4 displayed thick packages of brecciated diorite, which are shown to continue to depths of up to several hundreds of metres, as was observed in drill hole TLG-ZJ419-105. The sequence is interpreted as the northern continuity of the Z87 brecciated diorite sequence.

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 lithogeochemistry 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 polyphasic 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 7-8: 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.

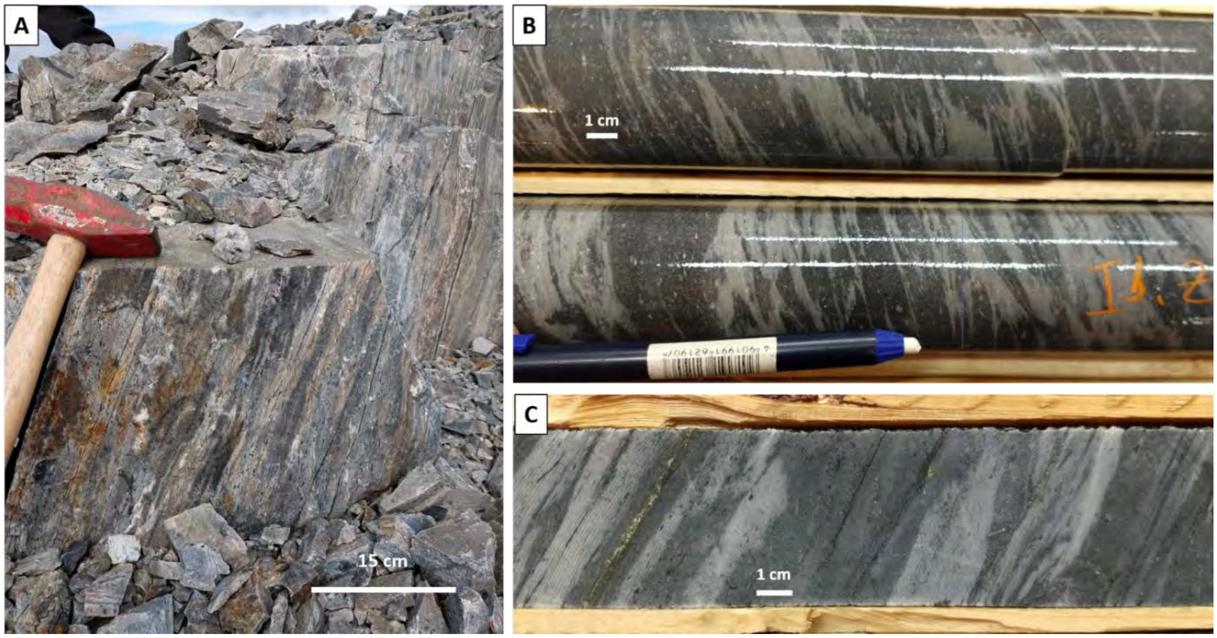
B. Typical mineralized brecciated diorite in a drill core; porphyritic diorite fragments in an amphibole-biotite-rich matrix

Felsic Dykes

Felsic dikes 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 dikes vary from massive or aphanitic to phaneritic and strongly foliated depending on the amount of sericite (Figure 7-9).

Two main decameter-thick felsic dikes occur at Z87, comprising the footwall and hanging wall of the main mineralized zone. In the J zone, the felsic dikes occur mainly in the immediate hanging wall of the mineralized diorite, are discontinuous, and occur in an anastomosing pattern, up to ten metres thick. The Z87S zone is dominated by felsic dikes, up to several metres thick, occurring in an anastomosing and locally stockwork-like pattern.

Figure 7-9: Photographs; Showing Diorite and Brecciated Diorite



Source: Troilus (2019)

A. Felsic dikes in outcrop, 87 pit; massive to slightly laminated

B. Porphyritic felsic dike showing sericite alteration overprint; apparently transposed by the main foliation, Zone 87 South

C. Mineralized massive felsic dike showing silicification and sericite alteration, Zone 87 South

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 J4 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, but the foliation is observed to wrap around the competent granitic bodies at the regional scale. This suggest the granite bodies were emplaced during the formation of the foliation in a late- to post-tectonic timing. A preliminary U/Pb age date of 2698 Ma was determined for titanite from the Parker granite (Goodman et al., 2005).

7.2.2 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.

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.

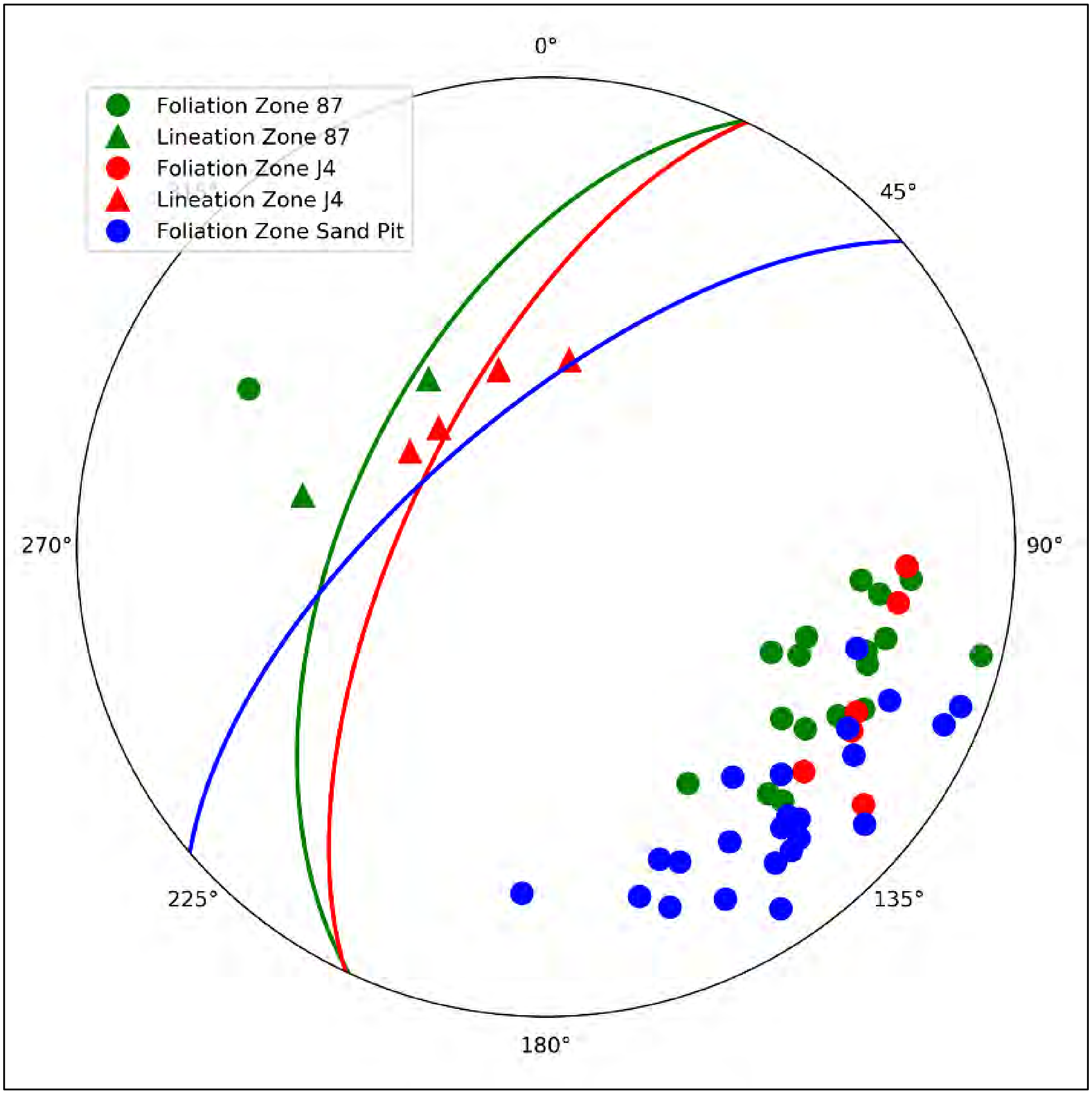
A down-dip stretching lineation oriented -60 322° /° within the foliation is observed to affect

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.

Stereonets of main planar and linear structures at Troilus are shown in Figure 7-10.

Figure 7-10: Stereonet of Main Planar and Linear Structures



Source: Modified from SRK (2018)

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, Southwest, 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).

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.

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.

7.3 Mineralization

The main mineralized zones at the Troilus Property occur around the margins of the Troilus Diorite, and comprise the Z87 Zone (including Z87S), and the J4/J5 Zone. Other important mineralized zones discovered to date include the northern continuity of the J4/J5 Zone, named the Allongé Zone, and the southwestern margin of the metadiorite (including the Z86 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.

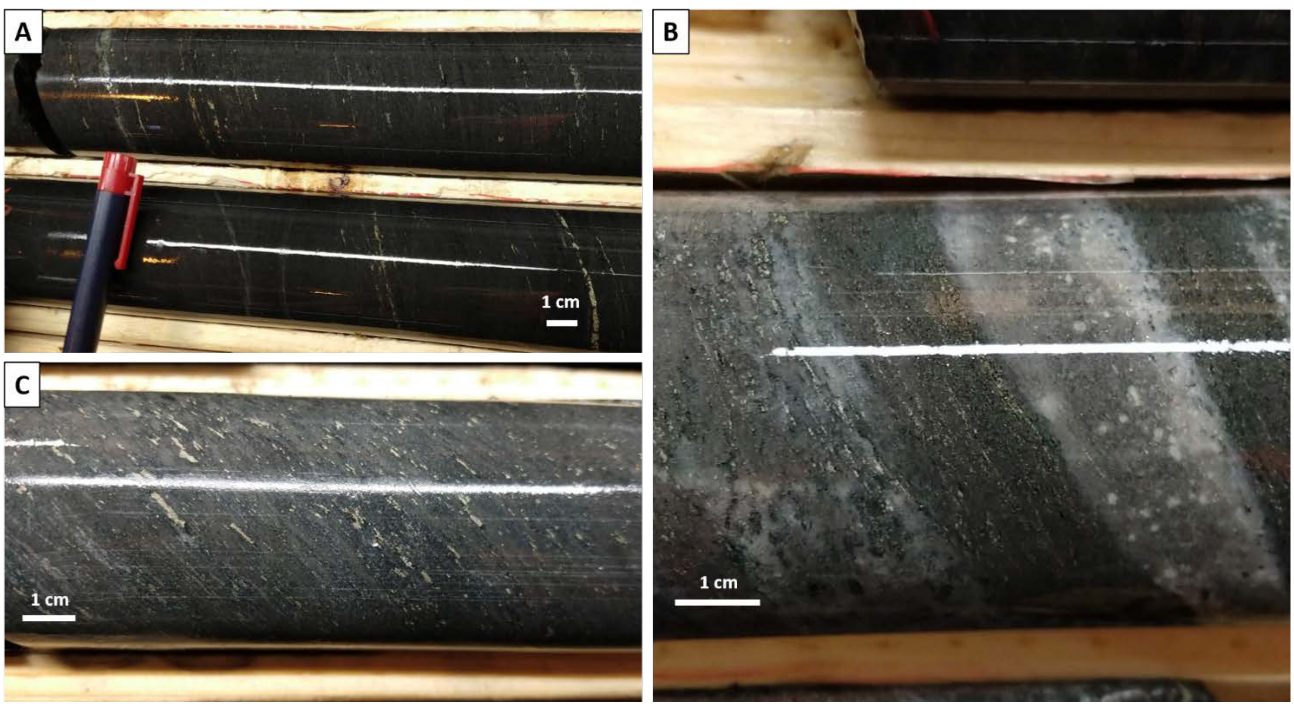
TYPE 1 – Disseminated Mineralization

Disseminated mineralization comprises the majority of the deposit's copper content (>90%, Goodman et al., 2005), particularly in the Z87. 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 wt. % and 5 wt. % of the rock. The most abundant sulfides are pyrite, chalcopyrite, and pyrrhotite.

Gold occurs as fine grains of electrum, up to 20 µm wide along sulfide grain boundaries, and filling fractures within sulfide grains, containing up to 15 wt. % Ag (Goodman et al., 2005).

At Z87, the mineralization is developed within an amphibolitic unit and the brecciated unit, located between the two thickest felsic dikes (Goodman et al., 2005), and it is coincident with a zone of strong biotitic alteration.

Figure 7-11: Photographs; Showing Disseminated Mineralization



Source: Troilus (2019)

A. Disseminated pyrite in a fine grained, biotite-rich, diorite - J4 zone

B. Brecciated Diorite; fine sulfides disseminations in the amphibole-biotite-rich matrix - 87 zone

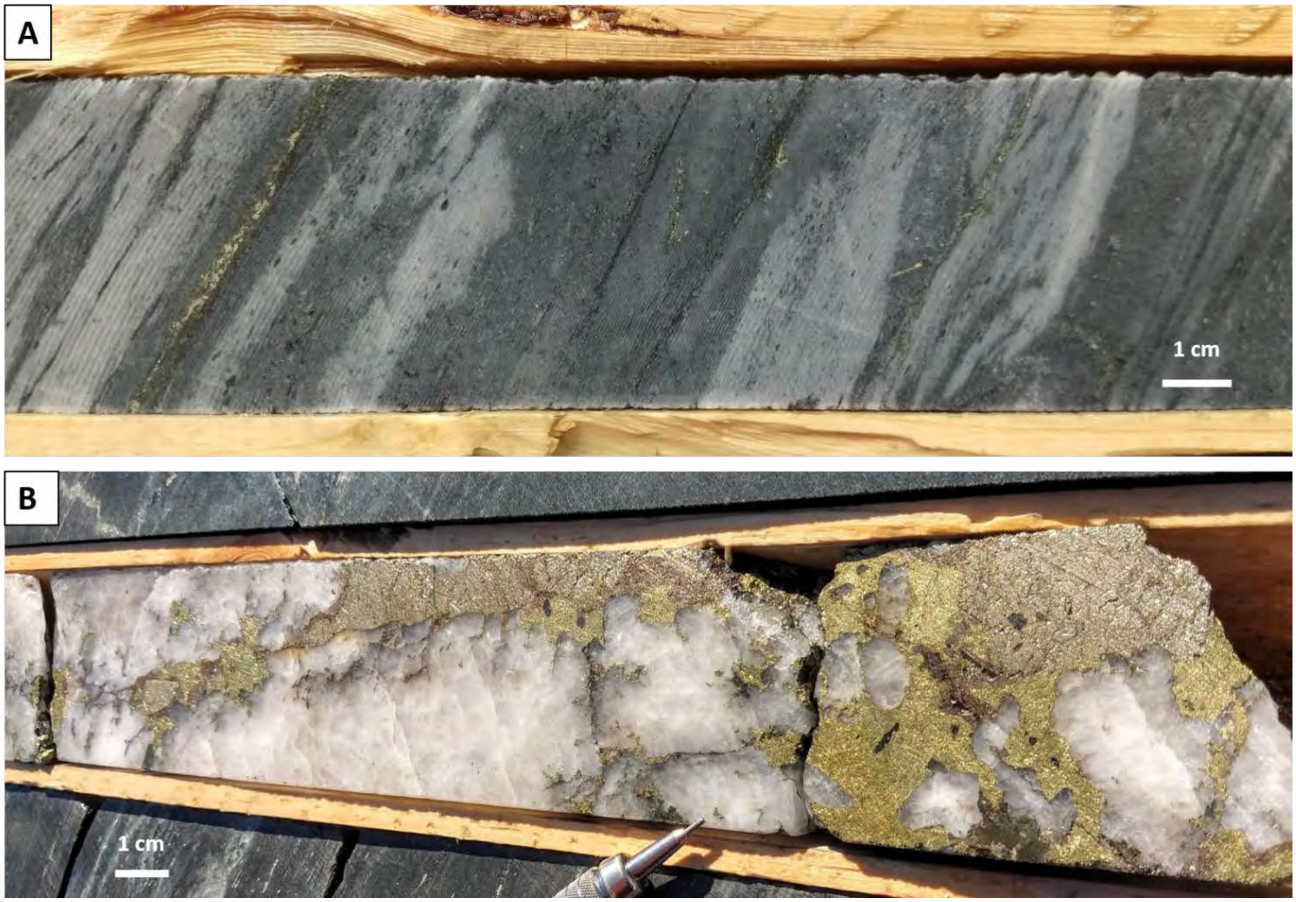
C. Disseminated medium grained pyrite in volcanic laminated rock - J5 zone

TYPE II – 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.

Several generations of gold-bearing veins have been identified and described by Goodman et al. (2005), and Larouche (2005), the latter especially focused on J4 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. 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. Locally, a second set of gold bearing quartz veinlets cut the first. These carry fine grained gold (>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 7-12: Photographs; Showing Vein Hosted Mineralization



Source: Troilus (2019)
Millimetric Py-Po-rich veinlet in an altered felsic dike (sericitization and silicification) - Z87 South
B. Atypical very high grade quartz veins, up to over 1-m thick; remobilized pyrite-chalcopyrite-pyrrhotite - J zones

7.3.1 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.

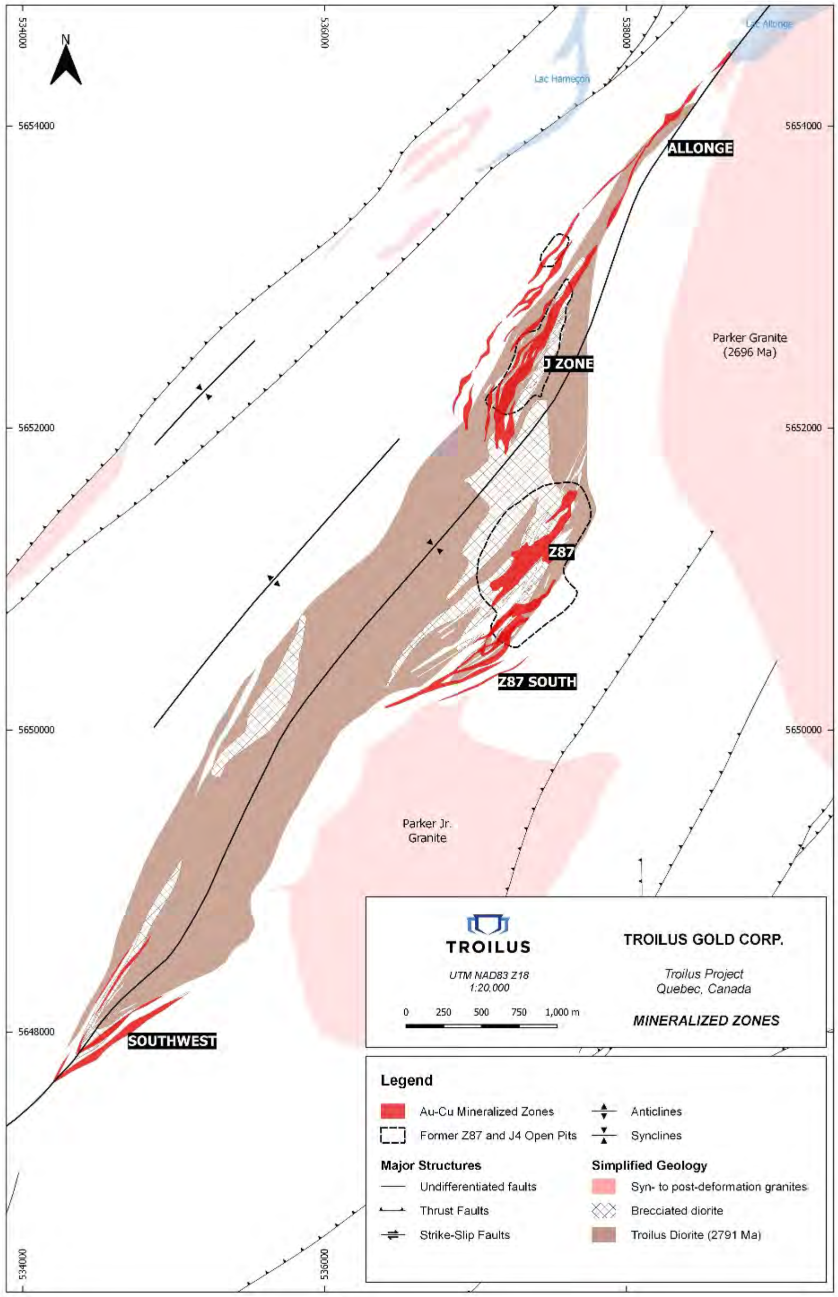
Calcic Metasomatism

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.

7.4 Mineralized Zones

There are four main deposits that make up the Troilus Gold Project: Zone 87, Zone 87 South, J Zone and SW Zone (Figure 7-13)

Figure 7-13: Main Mineralized Zones



Source: Troilus (2019)

7.4.1 Zone 87

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 -30° to -50°. The north and south extensions of Z87 "horsetail" out into narrower branches of mineralization. Two branches are well defined in the north, whereas three branches are less defined to the south.

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 it is gold-rich relative to copper.

7.4.2 Zone 87 South

Z87S is located directly southwest of the main former open pit mine, Z87. The two zones are separated by a felsic dyke and a zone of intense deformation dipping at 45° to 55° northwest. Z87S itself dips of ~50° northwest. This angle suggests Z87 and Z87S may merge at approximately 450 m below surface. The presence of a gold rich interval below Z87 in borehole TLG-Z8718-002 is probably the expression of Z87S at depth.

The 2019 drill program in Z87S was designed to follow-up on the positive few holes drilled in this zone in 2018. The new results have outlined extensions of mineralization to the south and down-dip of the previously known mineral envelope in Z87.

The mineralization at Z87S is visually comparable to what is seen in the main zone of Z87, however the geology can be characterised as more felsic (silicic) alteration and is distinctly transitioning into a unit of massive sulphides (primarily pyrite with chalcopyrite) in the footwall. A preliminary geochemical study of Z87S has a recognizable base metal signature that is unique to this area. This zone also exhibits the same structural pinch and swell nature of mineralization as the other main mineralized zones.

The host rock of that sulphide- rich zone is characterized by and intermediate to mafic volcanic unit similar to the sulphides rich zone of the hanging wall of J4 corresponding to the south-western extension of the J5 zone.

7.4.3 J4/J5 Zone

The J Zone orebody hosts two mineral zones: J4 and J5. J4 is the smaller of the two formerly mined open pits along with the main Z87 zone. The ore bodies in the J4 zone are hosted in the northern continuity of the Troilus Diorite and, similarly to what is observed in the main zones Z87 and Z87S, are elongated parallel to a penetrative northeast trending foliation, moderately to steeply dipping to the north west.

From top to bottom, the sequence comprises (i) a volcaniclastic unit, occurring along the hanging wall of the mineralization, and composed of well laminated intermediate to felsic rocks, locally mineralized, with semi-massive sulfide occurrences; and (ii) a thick metadioritic unit, comprising fine to coarse grained diorites that are locally brecciated. They are commonly crosscut by decametric to metric-scale felsic dikes, which are mostly concentrated in the upper parts of the sequence, in the immediate hanging wall of the mineralized intervals. Towards the bottom of the sequence, in the footwall, typical diorite breccias are present, displaying intense silicification and being locally importantly mineralized.

The main mineralized intervals in the J4 zone are characterized by sulfide stringers and fine sulfide disseminations along the foliation occurring within a very fine grained biotite-rich and silicified diorite. Pyrite is the main sulfide, and it is intrinsically associated with gold mineralization.

Results from hole TLG-J419-092 extended the limits of the gold-rich mineralization outside of the known mineral resource envelope both at depth and to the east. This zone located in the footwall of the main gold zone of J4 is characterized by a far less deformed texture than typical J Zone mineralization with clear brecciation and disseminated sulphides within the recognizable Troilus Diorite was identified in the stratigraphic footwall.

Compared to Z87, the J4 Zone has a lower copper grade, more free gold, and dips more steeply at -65°. J4 extends for approximately 1,200 m from 14,100N to 15,300N and is approximately 200 m wide from 9,500E and 9,700E. Individual mineralized shoots plunge steeper to the north. The north half of J4, from approximately 14,600N, contains one main corridor of mineralization, which is 20 m to 50 m in horizontal width. Grade-contoured blasthole data reveal the presence of closely spaced lenses, which strike to mine-grid northeast and dip towards mine-grid northwest. These lenses are located within and extend beyond the interpreted mineralized envelope limits. In the southern half of J4, three main lenses of generally lower grade and more diffused gold mineralization have been identified. The mineralization here averages approximately 100 m in horizontal width with intervening waste.

7.4.4 Southwest Zone (SW Zone)

The SW Zone is situated approximately 3 km southwest of the Z87 Zone. The current interpretation, based on recent drilling, is that the SW Zone appears to be the nose of a synclinal fold with a gentle plunge to the northwest (Figure 7-14).

As observed in all main mineralized zones on the Property, the SW Zone lithological sequence is comprised by a dominantly mafic footwall volcanic sequence, and a more intermediate to felsic hanging wall (Figure 7-14). This volcanic package is intruded by syn-volcanic dioritic and felsic rocks. Mineralization mainly associated with diorites, brecciated diorites, and felsic rocks. The SW Zone is located within the hinge zone of the interpreted Troilus Syncline, in a zone of tight folding. It has been divided in two distinct structural domains:

- the eastern domain, named the “Main Zone”, which hosts the largest part of the mineralized horizons, and received most of the drilling executed so far
- the western domain, which shows a narrower mineralized horizon, yet to be detailed drilled

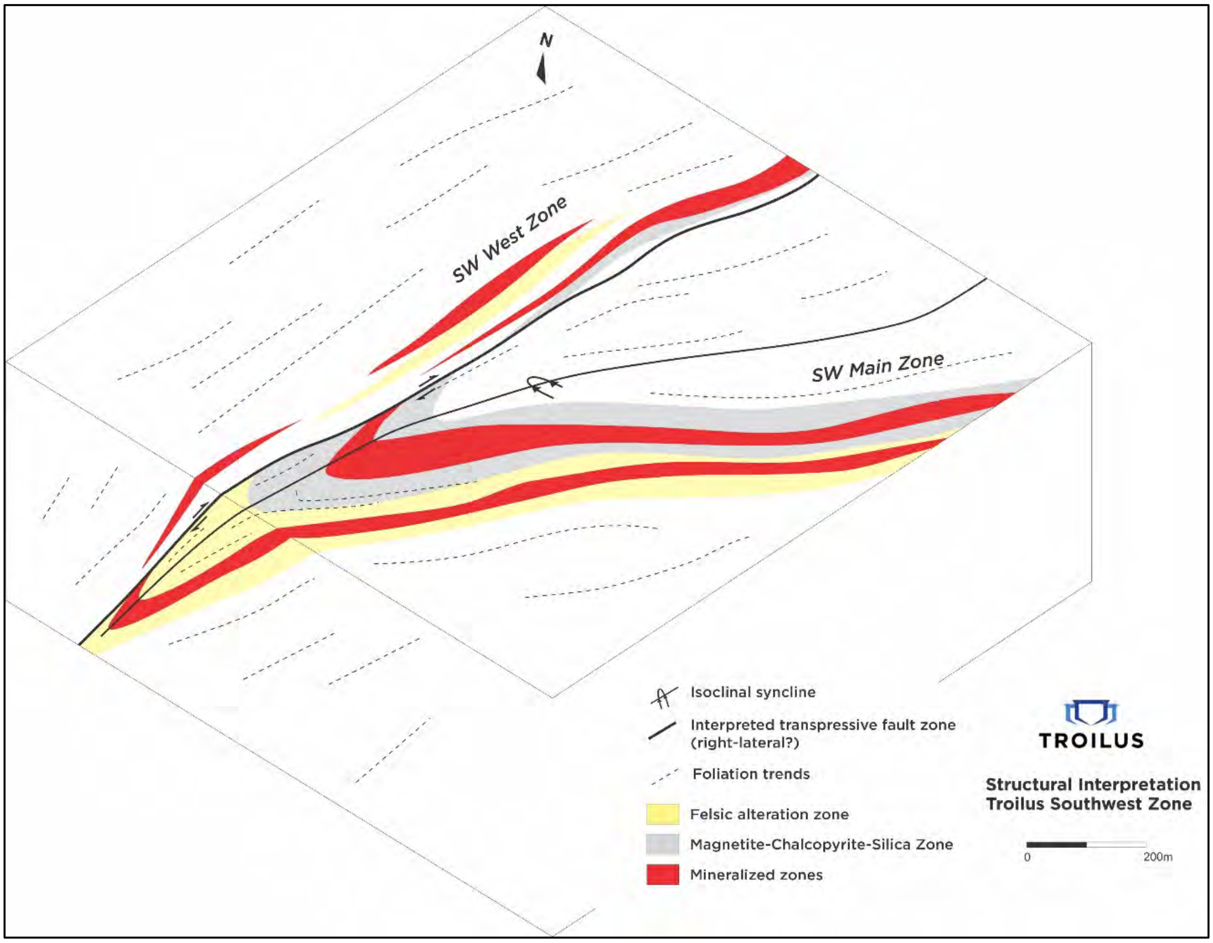
The Eastern Domain, or Main Zone, dominantly strikes ENE and comprises the eastern limb of the interpreted syncline. The Western Domain clearly offset the eastern portion, striking slightly more NE.

A major strike-slip shear zone is interpreted to have overprinted the folding system and characterizes a northeast dominant structure parallel to the fold axis, as can be observed in the local geological map and schematic block model (Figure 7-15). This shear zone is interpreted to be parallel to the main bedding and foliation, dipping to southwest. This structure is well marked by the geological distinction between east and west domains, as well as by a clear distinct strike angle of both limbs.

Figure 7-14: Geology Map of the SW Zone; showing drill hole traces

Source: Troilus (2020)

Figure 7-15: Geology Map of the SW Zone; showing drill hole traces



Source: Troilus (2020)

The footwall mafic volcanic sequence in the Southwest 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) feldspar porphyry and (ii) felsic dikes. They share similar compositional and textural characteristics and are often mistaken due to the lithological similarities and alteration pattern. Both Felsic dikes and 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 dikes are thinner and occur as “arrays” of several “dikes”, cross cutting the sequence, and often concentrated in the contact zone between mafic footwall and more intermediate to felsic hanging wall.

The feldspar porphyry defines a continuous, with tens of meters thick unit, occurring immediately above the mafic footwall sequence. It hosts an important part of the mineralization found in the eastern domain of 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 of the feldspar porphyry unit.

A magnetite-rich and highly silicified brecciated unit represents the main host rock for gold and copper mineralization at the SW Zone and occurs within typical fine-grained, locally porphyritic diorites. The original textures and composition have completely been replaced by an intense silica alteration. The brecciated texture is characterized by dark grey, highly silicified fragments- or pseudo-fragments, occurring in a chalcopyrite- pyrite- and magnetite-rich biotitic "matrix".

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 10-20 meters thick.

Fine-grained, porphyritic diorites 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 the Z87 show important similarities in terms of host rocks, mineralization style and geochemistry, as summarized below:

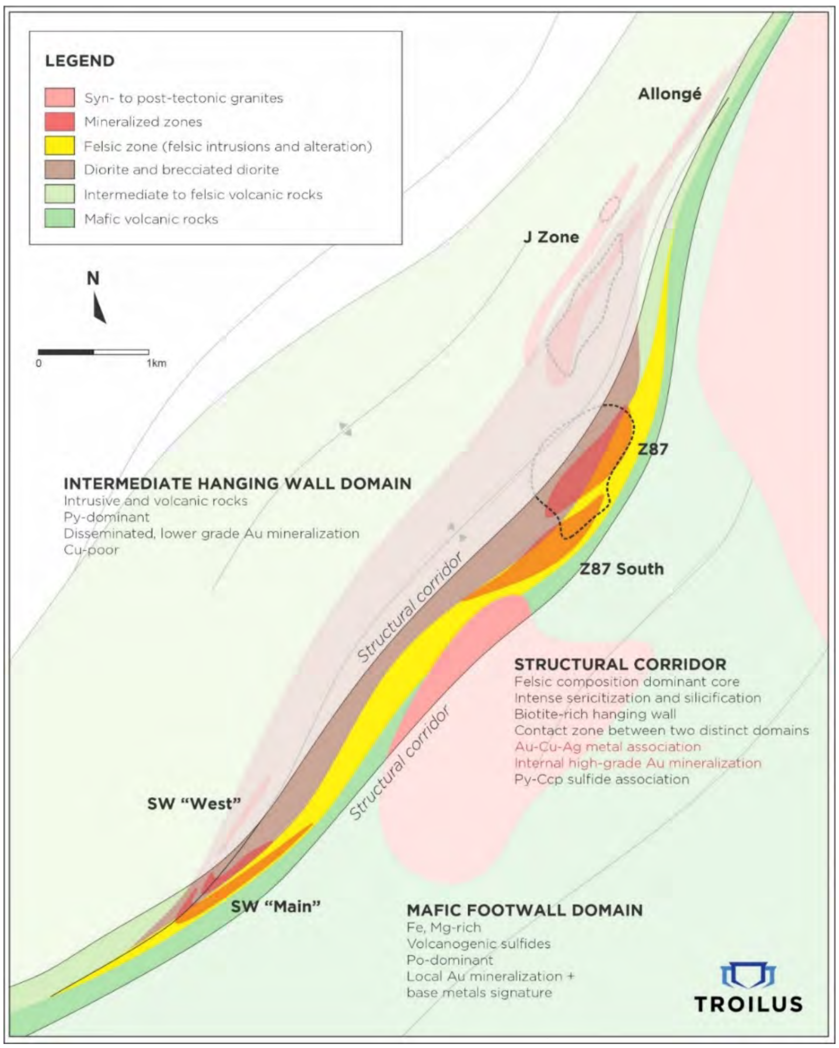
- similar Au-Cu-Ag metal association
- high-grade Au associated with chalcopyrite (filling micro-fracturing and in sulfide margins)
- zoning: py-rich hanging wall, py-Ccp core zone, py-Po footwall
- main host rocks:
  - brecciated/pseudo-brecciated upper ore body: higher grade, Au-Cu association
  - felsic unit/alteration: bottom ore body, lower grade, relatively Cu-poor; ("felsic dike" with porphyritic textures at Z87 & Feldspar Porphyry at SWZ)
- least altered, medium to coarse-grained typical diorite in the hanging wall
- mafic, amphibole-rich, volcanic sequence in the footwall

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 intrusive package at the hanging wall (Figure 7-16).

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

Figure 7-16 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 7-16: Simplified Geology Map of the Mineralized Zones on the Property; highlighting the structural corridor



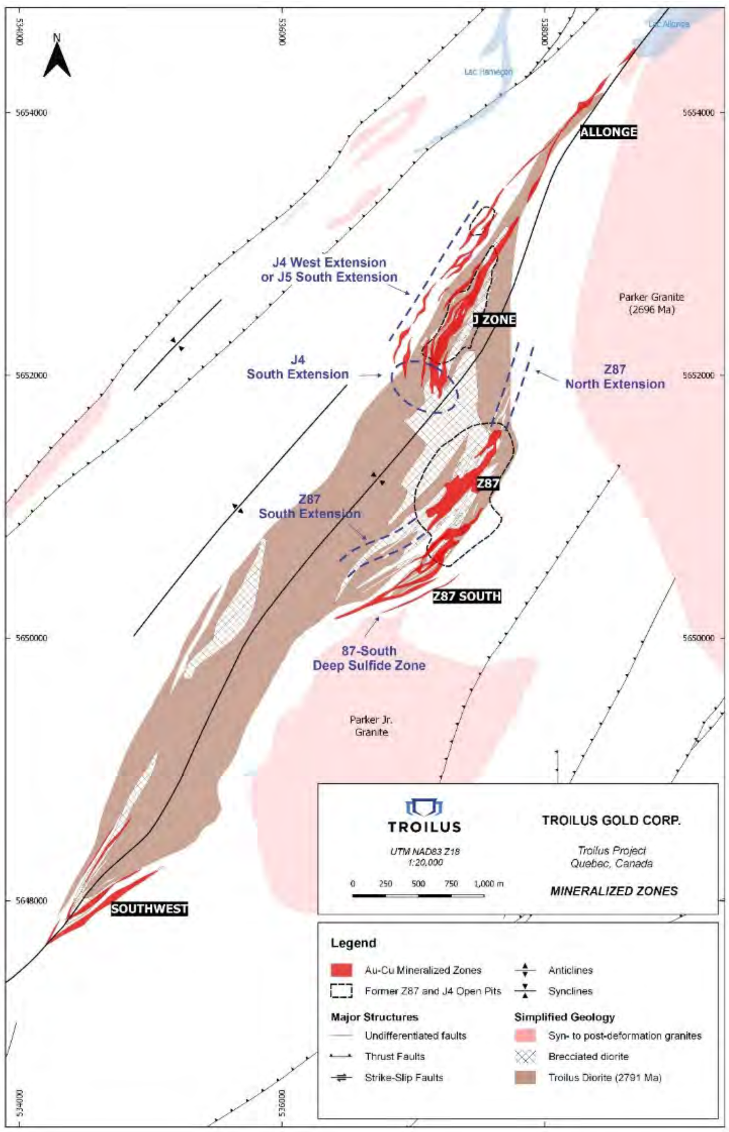
Source: Troilus (2020)

7.5 Prospects/Exploration Targets

This section details lithological and structural particularities of the following potential mineralized extension zones (Figure 7-17):

- Z87 North Extension
- J4 South Extension
- J4 West Extension (J5 South Extension)
- J4/J5 Zone North Extension (Allongé Zone)

Figure 7-17: Exploration Targets on the Property



Source: Troilus (2019)

7.5.1 Z87 North Extension

The northernmost current borehole at Z87 is TLG-Z8718-044W. The mineralized horizon is intersected near the bottom of the hole. The intersection includes 10.58 g/t Au equivalent over two metres, and 7.82 g/t Au equivalent over six metres (Troilus Gold press release, October 31, 2018), and could correspond to the downdip extension of a gold trend present at surface. The intersection of gold in TLG-Z8718044W opens the potential for an extension of the gold mineralization to the north.

Near Surface Potential: Most of historical holes drilled at the northern extension of Z87 Pit are enriched in gold. Holes KN 38, 39, 46, 101, 102, 139, 376, 397 and 398 show gold values that can be traced up to one kilometre and follow a 360° trend.

Deep Potential: Hole TLG-J419-105, drilled from the hanging wall of J4, confirmed that the same geological sequence from Z87 can be trace as far as 650 m to the north of the Z87 Pit at a vertical depth of 750 m.

7.5.2 J4 South Extension

The Zone J4 gold trend at surface bends to the south and toward Z87. In TLG-ZJ418-076 and 083, the best auriferous intersections are located at the bottom of the hole, which validates the interpreted change of direction of the main gold trend. The main potential for the extension of Zone J4 to the south lies in the area between J4 and Z87, however, the potential for gold mineralization is also open to the south-southwest and to the west.

7.5.3 J4 West Extension (J5 South Extension)

The results of the 2019 drilling program have significantly extended the boundaries of known mineralization at depth from the northeast to the southwest in the J4 Zone, well beyond the formerly mined J4 pit. The shallower intercepts of most holes are believed to be mineral extensions from the neighbouring J5 mineral zone. This is further evidence that suggest that J4 and J5 zones may prove to be one and the same. The J4 and J5 zones remain open at depth.

7.5.4 J Zone North Extension (Allongé Zone)

In October 2018, Troilus began a preliminary surface exploration program focused on applying its newly developed structural and geological model regionally to the Troilus belt. A total of 172 samples were collected from 157 outcrops and were sent for assay. Results have defined a clear extension of mineralization from J Zone over a strike length of 1.8 km extending from the edge of the J Zone to the northeast. Prospecting and mapping have identified additional gold-bearing mineralization located along the north-easterly strike projection of the J Zone. These newly discovered units, paired with minimal local historic drilling, have opened the potential to expand the Troilus deposit to the northeast.

Highlights:

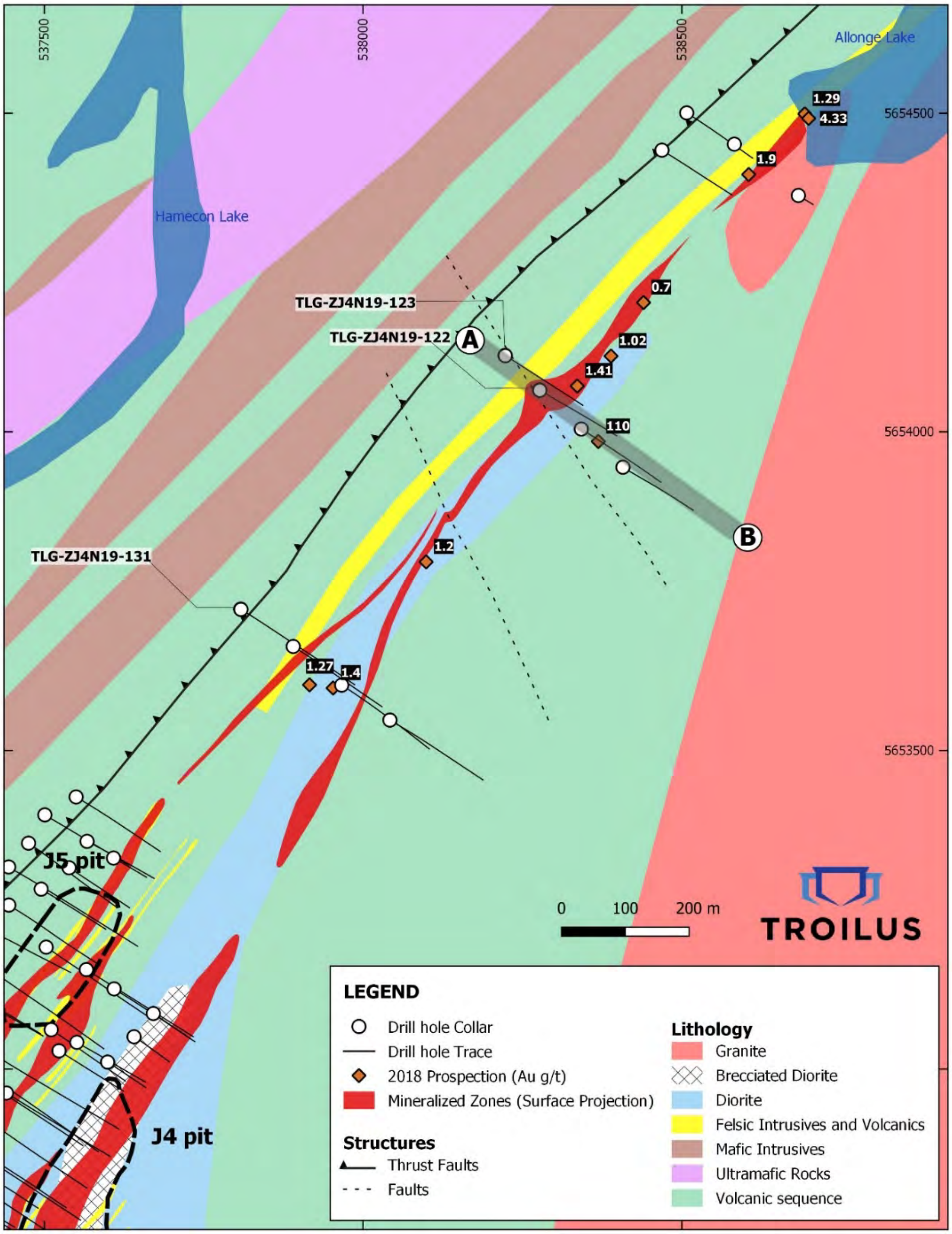
- 110 g/t Au (visible gold) from rock grab sample approximately one kilometre along strike from J4 open pit hosted in foliated diorite, the same host rock as the J Zone
- 4.33 g/t Au, 1% Cu, and 49.5 g/t Ag from channel sampling located directly adjacent to the Troilus North and 1.8 km northeast of J Zone

- 1.9 g/t Au, 0.3% Cu and 16.3 g/t Ag hosted in altered rhyolites from a grab sample located directly southwest of 4.33 g/t channel sample
- 1.4 g/t Au, 0.6% Cu, and 10.3 g/t Ag from channel sampling less than 400 m from the northeast limit of J4 open pit

Based on the positive 2018 surface results, an exploration drilling program was carried out in 2019, with a total of 12 DDHs on three sections spaced 500 m apart. The new results have successfully confirmed the potential to extend the J Zone mineralization to the northeast.

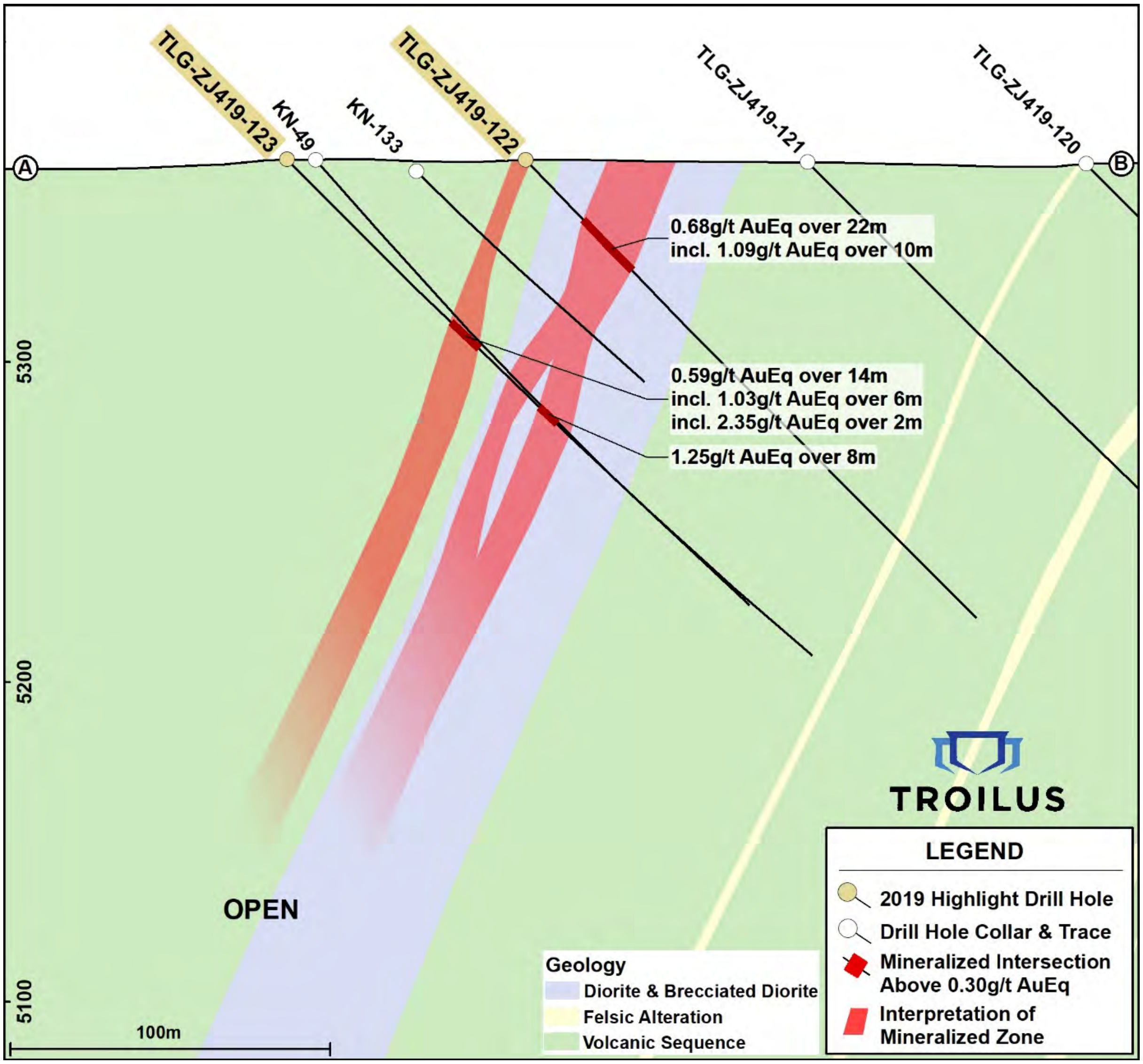
The mineralized intervals in the Allongé Zone lie within the same geological sequence present in the hanging wall of the J zones, comprising a finely laminated intermediate to mafic volcanic sequence, locally intercalated with rhyolite metric layers, and the Troilus Diorite, which continues north up to the southern margin of the Allongé Lake (Figure 7-18 and Figure 7-19). Gold mineralization is observed to be associated with pyrite-rich millimetric-scale layers and stringers that are oriented parallel to a penetrative northeast foliation, occurring both within the diorite and the volcanic sequence.

Figure 7-18: Geology Map, Allongé Zone; showing 2019 drill holes



Source: Troilus (2019)

Figure 7-19: Cross-section 16,500N, Allongé Zone; looking northeast



Source: Troilus (2019)

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 Geological Characteristics Supporting the Proposed Genetic Models for the Troilus Deposits

Model	Timing	Host Rocks	Sulfides 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-peal 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, sp-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)

*modified from Diniz (2019)

Note: (py)-pyrite, (ccp)-chalcopyrite, (po)-pyrrhotite, (gn)-galena, (sp)-sphalerite

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 in terms of metal content, hydrothermal alteration, and host structures, are described in the Troilus deposit, similarly to what is observed in the examples discussed above. The combination of more than one style of mineralization can represent evidence of multiple stages of gold mineralization, in the cases discussed, an early magmatic-hydrothermal event followed by syn-deformational gold input and remobilization. However, these deposits represent well known and largely studied examples, while the Troilus deposit is still poorly understood, and most of the current interpretations lack clear evidence to determine whether the distinct styles of mineralization are different in age and nature or not (Diniz, 2019).

Based on this, and similarities with other known multi-stage ore systems in the Abitibi greenstone belt (e.g. Canadian Malartic, McIntyre), as well as other magmatic-hydrothermal deposits (e.g. Côté Gold), 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 exploration and development of the Project is described in Section 6. Since acquisition of the Project, Troilus compiled historical exploration and drilling data and carried out field mapping and prospecting programs. Additionally, Troilus has completed several drilling programs on the Project and are described in Section 10.

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 J4/J5 Zones. Compilation of drill hole data for holes drilled away from the Troilus deposit has shown that there are a number of holes with gold values greater than 200 ppb over ten metres. 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 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 drill holes 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.

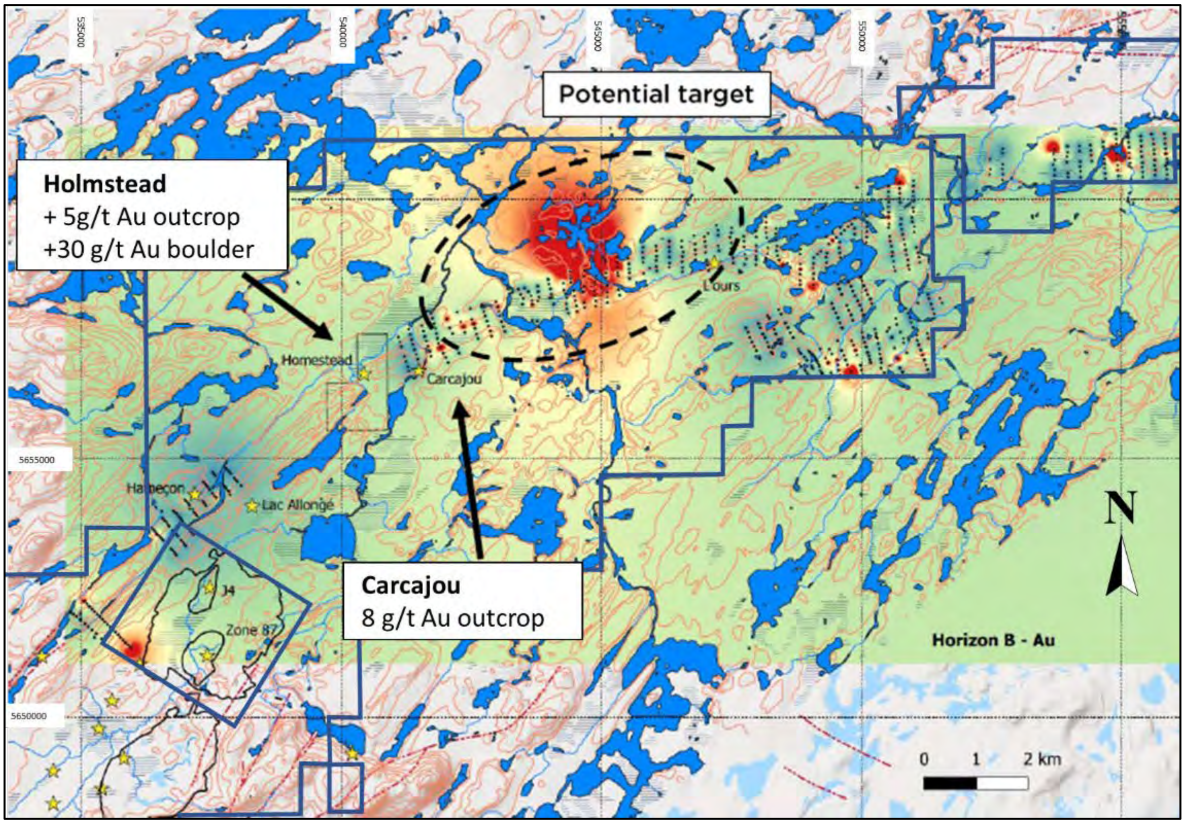
9.2 Troilus, 2018 – Present

Field mapping and prospecting work in 2018 and 2019 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.

The field exploration programs on the northeastern 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. The results of the soil geochemistry survey is shown in Figure 9-1 and Figure 9-2, for gold and base metals, respectively.

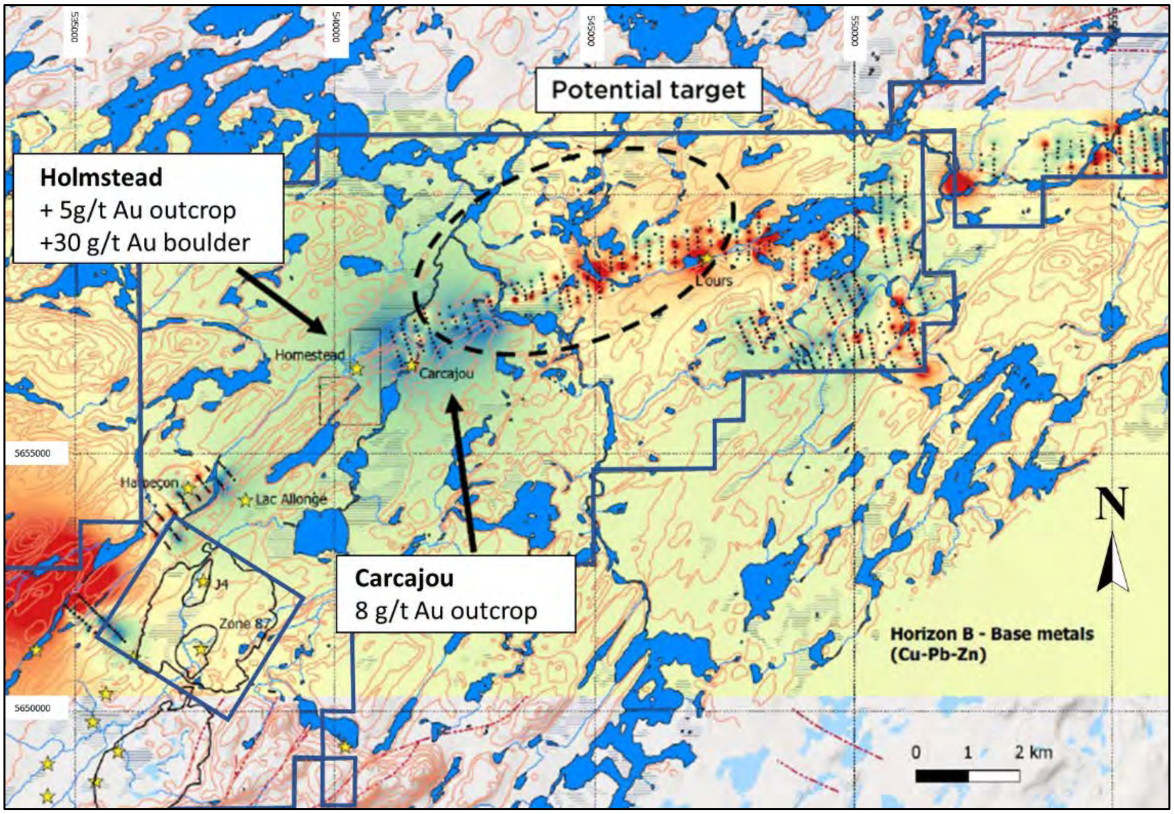
Surface exploration was also carried out over the Z86 Zone and SW Zone.

Figure 9-1: Gold in Soil Anomaly Map of the Property



Source: Troilus (2019)

Figure 9-2: Base Metal in Soil Anomaly Map of the Property



Source: Troilus (2019)

9.2.1 Structural Study, SRK (2018)

In early 2018, SRK Consulting (Canada) Inc. (SRK) was retained to conduct a structural geology investigation at the Project. The study focussed on the exposed geology in the Z87 open pit and the J4/J5 open pit. [include summary of findings; or check if already stated in Section 7]

9.2.2 Southwest Zone

The Sand Pit, discovered in 2018, is located at the southern limit of the Southwest Zone and is dominantly composed of an auriferous breccia intruded by a series of intrusions, including felsic dikes. A series of amphibolite outcrops are present to the southeast, and diorite (un-brecciated) is present to the northwest. The breccia and sulphides are strongly transposed, and some remnants of folds can be observed, which indicates a pre- to early-D1 emplacement of the sulphides. They are preferentially hosted in the breccia matrix. The felsic dikes are altered, however, with only minor crosscutting quartz-sulphide veins, while the host breccia contains good disseminated sulphides content. All the observations suggest that Troilus-style gold mineralization is present in the southwestern extremity of the Troilus diorite intrusion.

Channel sampling results obtained in the Sand Pit, associated with historical mineralized boreholes, and a well know favorable lithological and structural characteristics, confirm that the southern portion of the Troilus intrusion represents a prospective exploration target. Additional diamond drill holes have been planned to test the full extent of the zone.

In late 2019 to early 2020, Troilus completed a preliminary drill campaign on the Southwest Zone.

9.2.3 Allongé and Carcajou Targets

Two main volcanic sequences are present on the Property. Occurring mainly in the northwestern region of the belt is a sequence consisting of basaltic to andesitic amphibolitized lavas, locally pillowed, likely of tholeiitic to ferro-tholeitic affinities. These rocks typically display very little to no sulfide content, and little biotite and silica alteration.

The second major volcanic sequence, located south of the latter, is intruded by the massive Parker granitic intrusion, and is the same phase that hosts the Troilus diorite and mineralized occurrences in the southern portion of the corridor. This second volcanic series consists of volcaniclastic intermediate to felsic tuffs and lavas.

Overall the entire sequence exhibits strong pervasive silicification and biotite and/or sericite alteration and comprises the hosting lithological unit for main mineralization occurrences in the northeast of the Property.

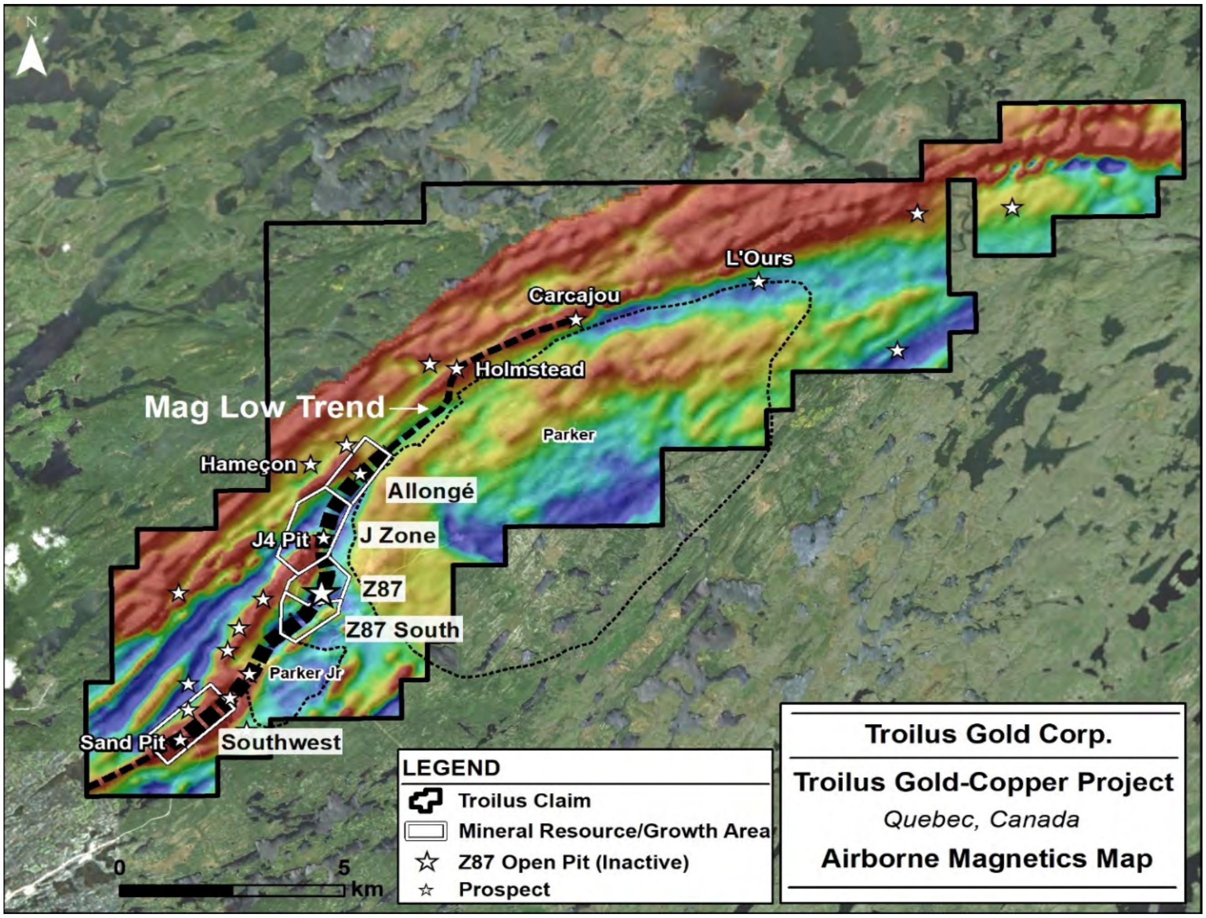
The Holmstead showing reported two grab samples over 5 g/t Au, situated one kilometre east of the north-northeast Lac Allongé. The Caracjou showing, situated four kilometres northeast and on strike with the Lac Allongé zone, reported a grab sample of 8 g/t Au. The mineralization consists of low content fine grained pyrite hosted in the felsic to intermediate volcanic rock, disseminated and stretched in the foliation, which is commonly observed in the J4 Zone.

Geophysical work and associated outcrop mapping show a general trend that hosts the Project that continues along Parker pluton (granite) to the east-northeast (ENE), of the Property. Recent mapping and data compilation demonstrate that potential for mineralized zones continue beyond the J4/J5 Zone.

Due to the size of the Property and limited exploration work, the area northeast of the known deposits remains open along a magnetic low trend which can be followed over 4.5 km from the J4/J5 Zone to the high-grade boulders outlined by Inmet in the 1980s (>10 g/t Au; Holmstead target), and over ten kilometres along the ENE trend (Figure 9 3).

Airborne magnetic geophysical surveys were completed in 2015 by High Sense Geophysics Ltd. (Toronto, Ontario based) for FQM, and in 2018 by Prospectair (Gatineau, Quebec based) for Emgold.

Figure 9-3: Airborne Magnetic Map of the Property



Source: Troilus (2020)

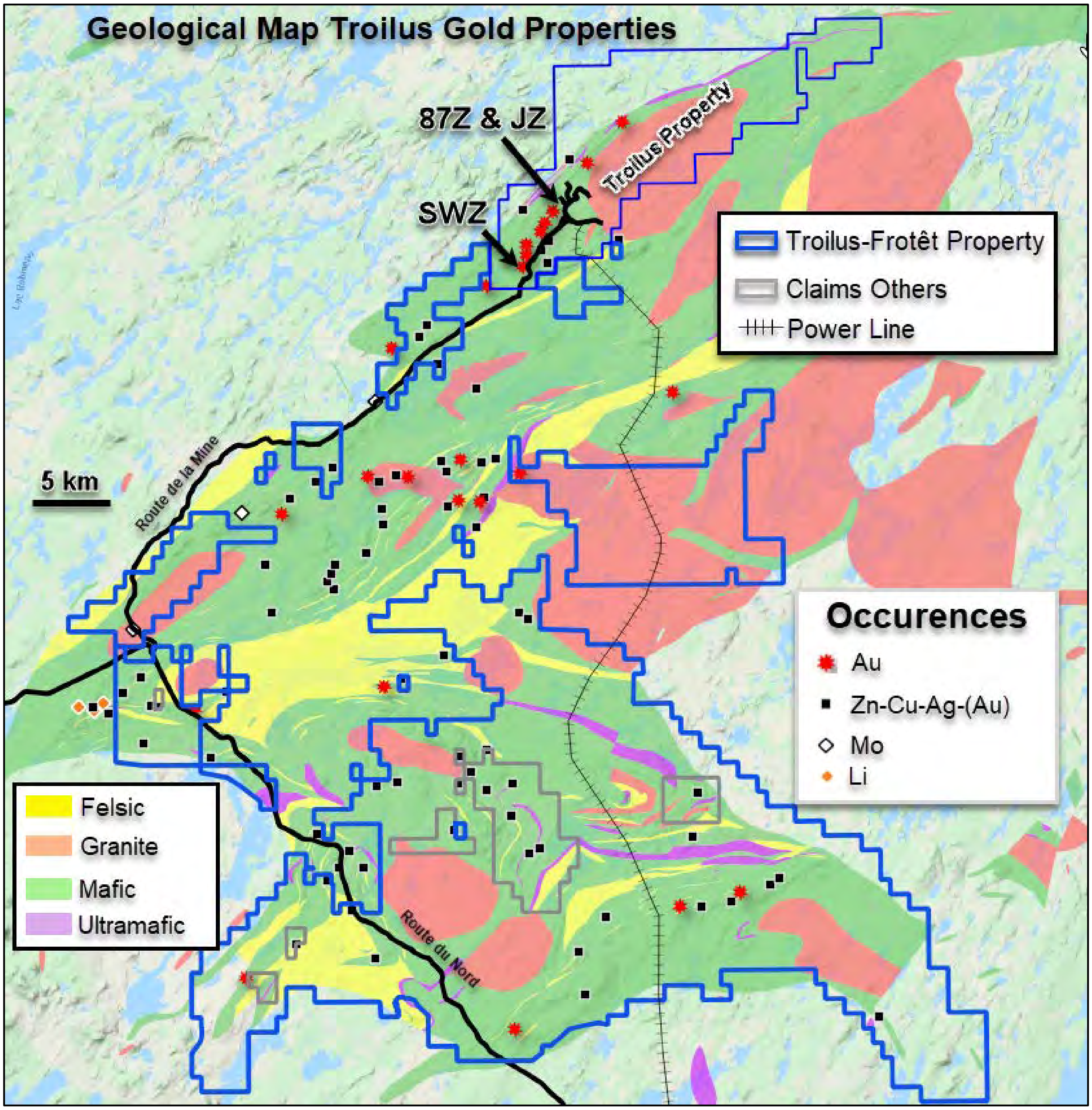
9.2.4 Troilus Frotêt Project

Following a major compilation of historical data, Troilus re-evaluated the potential of the entire Frotêt Domain by acquiring a major land position called Troilus-Frotêt Property (Figure 9-4).

Several types of mineralization are present on the Property. Based on Troilus' compilation work, two (2) main types of mineralization are dominant:

1. Volcanogenic Massive Sulphide (VMS)
2. Gold-Copper type, similar to the Troilus Mine

Figure 9-4: Troilus Frotêt Property; showing geology and mineral occurrences



Source: Troilus (2020)

In June 2020, Troilus completed a preliminary field exploration program applying a new regional structural and geological model, developed over the last two years, to the recently expanded Troilus-Frôtet Property. This property is situated to the south of the main mineralized zones of Z87, J Zones and the SW Zone. This led to the discovery of a previously unknown gold zone

During the summer 2020, Troilus completed an airborne high-resolution magnetic geophysical survey that covered 23,000 line km over the entire Troilus Frotet area (Figure 9-4). The airborne survey was carried out by Prospectair Geosurveys Inc., based in Gatineau, Quebec. The results and report of this survey is pending.

Beyan Gold Zone

Initial bedrock mapping and boulder tracing along the Route de la Mine North Block claims, situated approximately 8km southwest and along strike of the SW Zone led to the discovery of a new gold zone named Beyan Gold Zone (Figure 9-5).

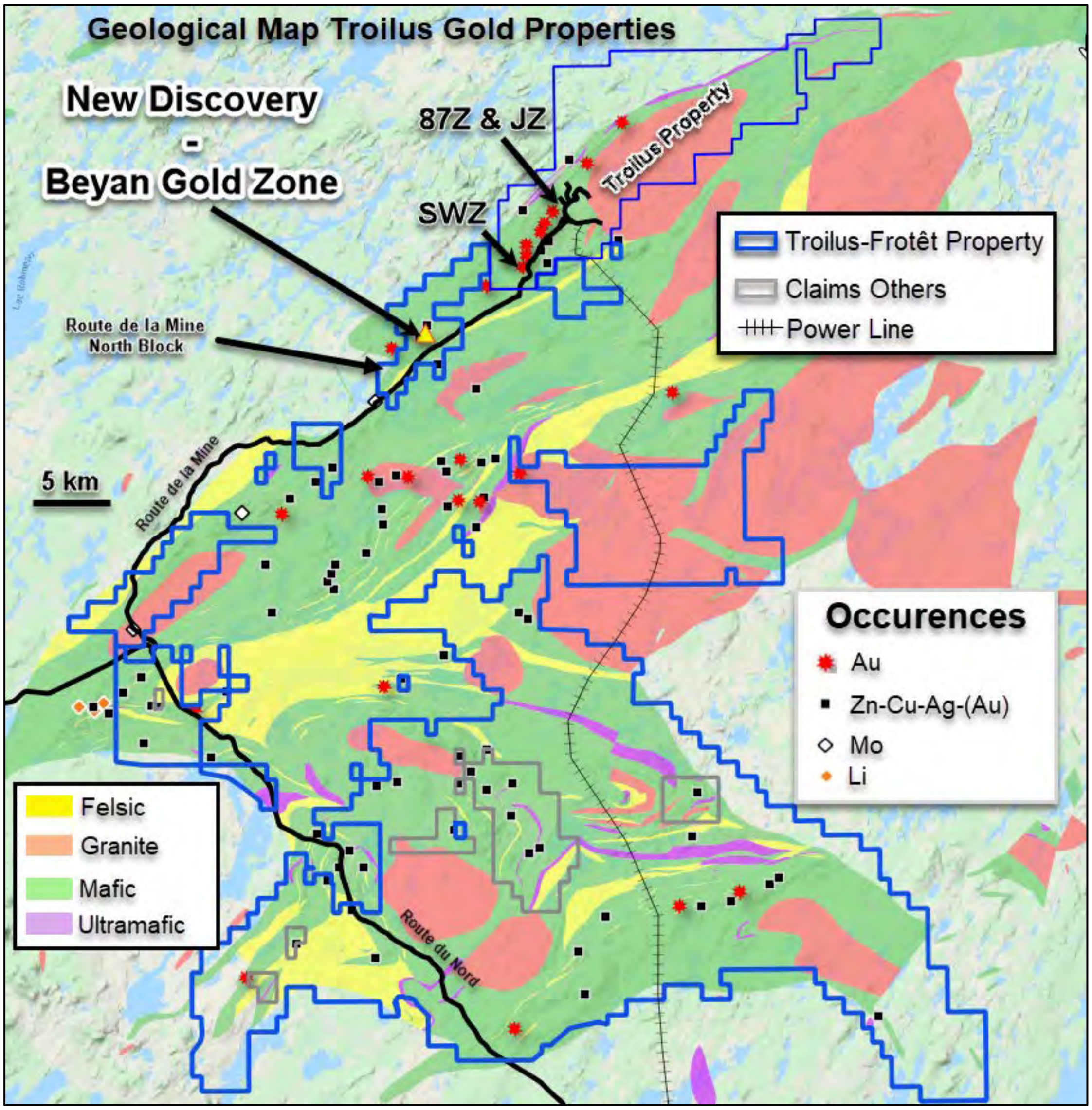
To date, 25 outcrop grab samples have returned anomalous gold values greater than 0.1 g/t Au with the best results returning 9.7 g/t Au and 32.5 g/t Ag (Table 9-1). A total of 14 grab samples from the Beyan Gold Zone have been collected from outcrop and can be traced on strike over 225 metres.

This new gold zone is part of a larger gold-bearing boulder field, identified by Troilus, characterized by several boulders that containing gold and silver values up to 2 g/t Au and 4.9 g/t Ag. Some of these boulders are large, between 1 m – 2 m in width, sub-rounded and sub-angular indicating a possible nearby source northeast of the new discovery. These boulders were found over a distance of 2.5 km. Results for approximately 50 samples are still pending from a total 150 samples that were collected in the vicinity of the Beyan showing.

On the Beyan Zone, the Troilus geological team collected over 600 channel samples from 71 channels. Results from these samples are also pending. Channel samples were collected by stripping the bedrock of overburden and marking the samples on the ground. Each channel is 4 cm wide by 10 cm deep cut and approximately 50 cm in length.

The main lithology hosting the gold and silver occurrences is amphibolite. This unit is highly deformed and strongly altered (silica, biotite, carbonate and ankerite). The amphibolite is commonly crosscut by smoky quartz veins and occasionally contains arsenopyrite. Intermediate to felsic units also contain gold. Moreover, a two metre thick banded iron formation (BIF) is in contact with the main mineralized zone. This horizon is unusual in this part of the greenstone belt and will be examined in more detail to determine its role during the process of the mineralization.

Figure 9-5: Troilus Frotêt Property; showing the Beyan Gold Zone, Geology, and Mineral Occurrences



Source: Troilus (2020)

Table 9-1: Summary of Initial Sample Results – Beyan Gold Zone

Sample No.	Type	Au (gpt)	Ag (gpt	Cu (%)	Zn (%)
Y937034	Outcrop	9.7	1.1	0.02	0.01
Y940321	Outcrop	2.8	1.2	0.01	0.02
Y937040	Outcrop	2.7	0.6	0.02	0.01
Y940702	Blocks	2.2	4.9	0.04	0.01
Y941731	Outcrop	2.1	0.6	0.01	0.01
Y940122	Outcrop	2.0	5.6	0.00	0.01
Y940311	Outcrop	1.7	32.5	0.09	5.77
Y941617	Outcrop	1.2	13.0	0.02	0.08
Y941619	Blocks	1.0	0.5	0.00	0.00
Y938846	Block	0.9	0.5	0.02	0.02
Y940322	Blocks	0.8	0.7	0.02	0.01
Y937002	Block	0.8	2.1	0.03	0.00
Y940128	Outcrop	0.7	0.7	0.01	0.01
Y940310	Outcrop	0.6	23.7	0.04	1.06
Y941732	Outcrop	0.4	<0.5	0.01	0.01
Y939237	Outcrop	0.3	<0.5	0.01	0.02
Y940318	Outcrop	0.3	<0.5	0.01	0.01
Y938123	Blocks	0.2	<0.5	0.02	0.01
Y938118	Block	0.2	4.9	0.13	0.10
Y939260	Outcrop	0.2	<0.5	0.01	0.01
Y940695	Blocks	0.2	<0.5	0.01	0.01
Y940134	Outcrop	0.1	<0.5	0.01	0.01
Y940701	Blocks	0.1	<0.5	0.02	0.01
Y940707	Blocks	0.1	0.5	0.01	0.01
Y940304	Block	0.1	<0.5	0.01	0.01

10 DRILLING

10.1 Drill 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 2020. Table 10-1 summarizes the diamond drilling programs completed on the property to date.

Troilus completed 91 drill holes totalling 37,510 m in 2018; 75 drill holes totalling 35,685 m from 2019; and 17 drill holes totalling 6,037 in 2020 (Table 10-1). Most of the 2018 and 2019 drill holes targeted Z87 and the J zones at depth and along strike. In the SW Zone, 24 drill holes were completed in (November 2019 to February 2020), totalling 8,500 m.

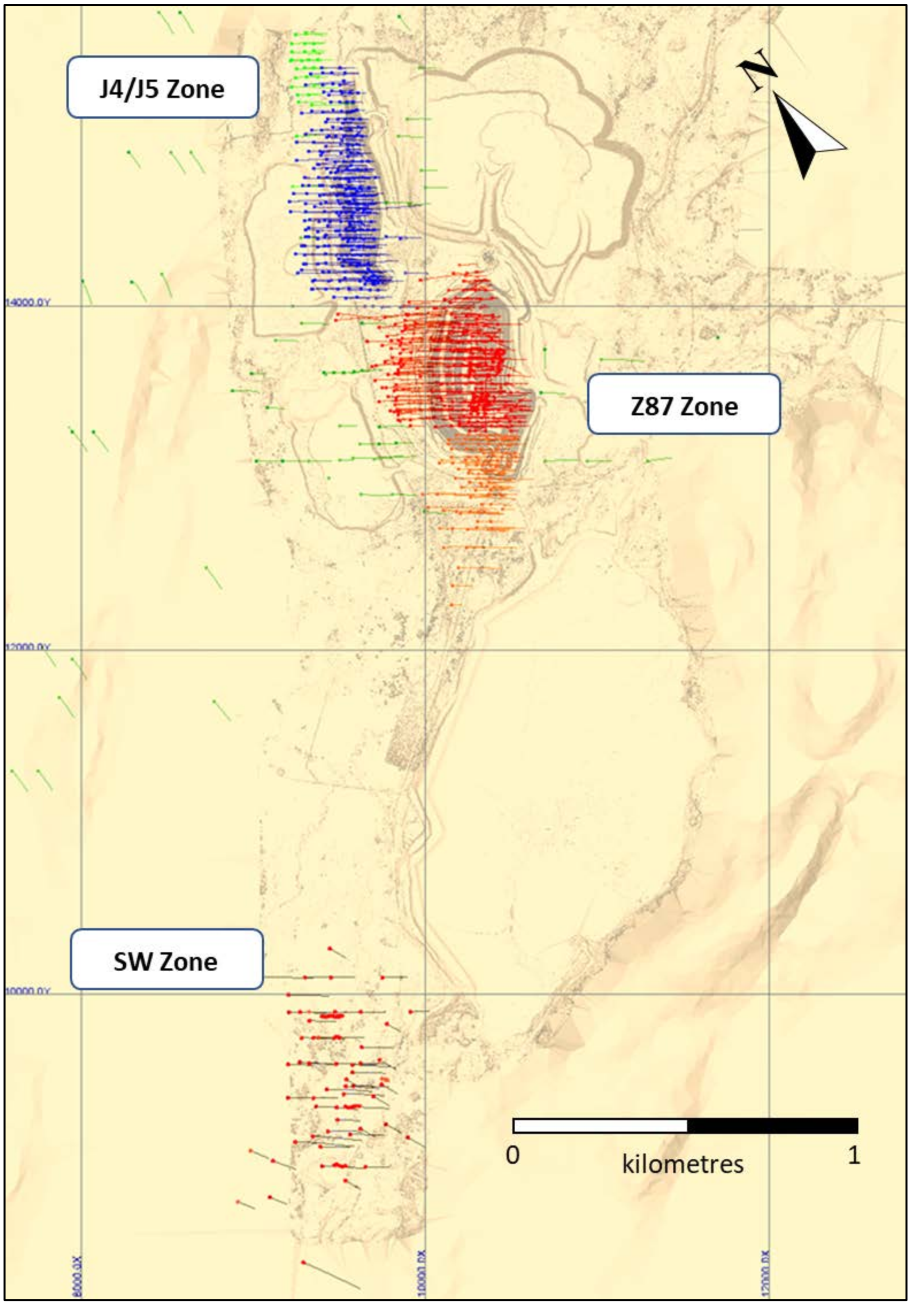
The current resource drill hole database contains 892 drill holes totalling approximately 215,347 m where the majority of the drilling targeted Z87, J4/J5 and SW Zones.

Table 10-1: Summary of Drilling

Year	Contractor	Core Size	No. Holes	No. Metres
1986-1989	Morissette Diamond Drilling	BQ (36.5 mm)	698	134,068
1990	Morissette Diamond Drilling	NQ (47.6 mm)		
	Benoit Diamond Drilling			
	Chibougamau Diamond Drilling			
1991-1993	Benoit Diamond Drilling	NQ		
	Chibougamau Diamond Drilling			
1995	Benoit Diamond Drilling	NQ ("KN" holes)		
	Morissette Diamond Drilling	BQ ("TN" holes)		
1997	Chibougamau Diamond Drilling	NQ ("KN" holes); BQ ("TN" holes)		
1999	Forages Mercier	NQ		
2000	Chibougamau Diamond Drilling	NQ (Z87 and J4 Zones); BQ (elsewhere)		
2002	Chibougamau Diamond Drilling	NQ		
2003-2005	Forages Mercier	NQ		
2007	Forages Mercier	NQ		
2018	Chibougamau Diamond Drilling	NQ	90	37,342
2019	Chibougamau Diamond Drilling	NQ	87	37,899
2020	Chibougamau Diamond Drilling	NQ	17	6,038

Figure 10-1 illustrates the drilling completed on the three mineralized zones of the Troilus Project. The red drill hole traces identify drilling in the Z87 Zone; blue traces in the J4/J5 Zone; and grey traces in the SW Zone.

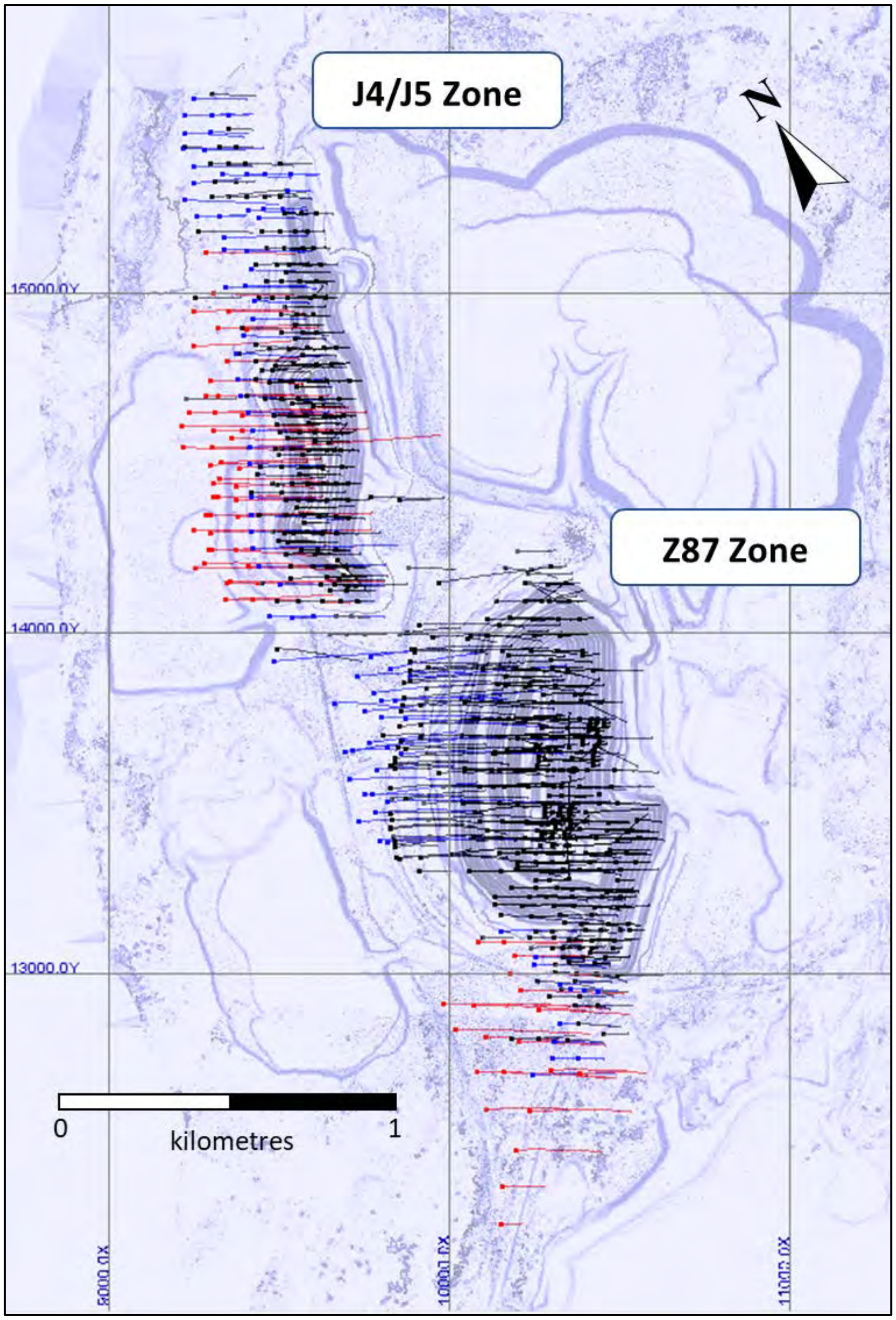
Figure 10-1: Drill Hole Map – Troilus Project



Source: Troilus (2020)

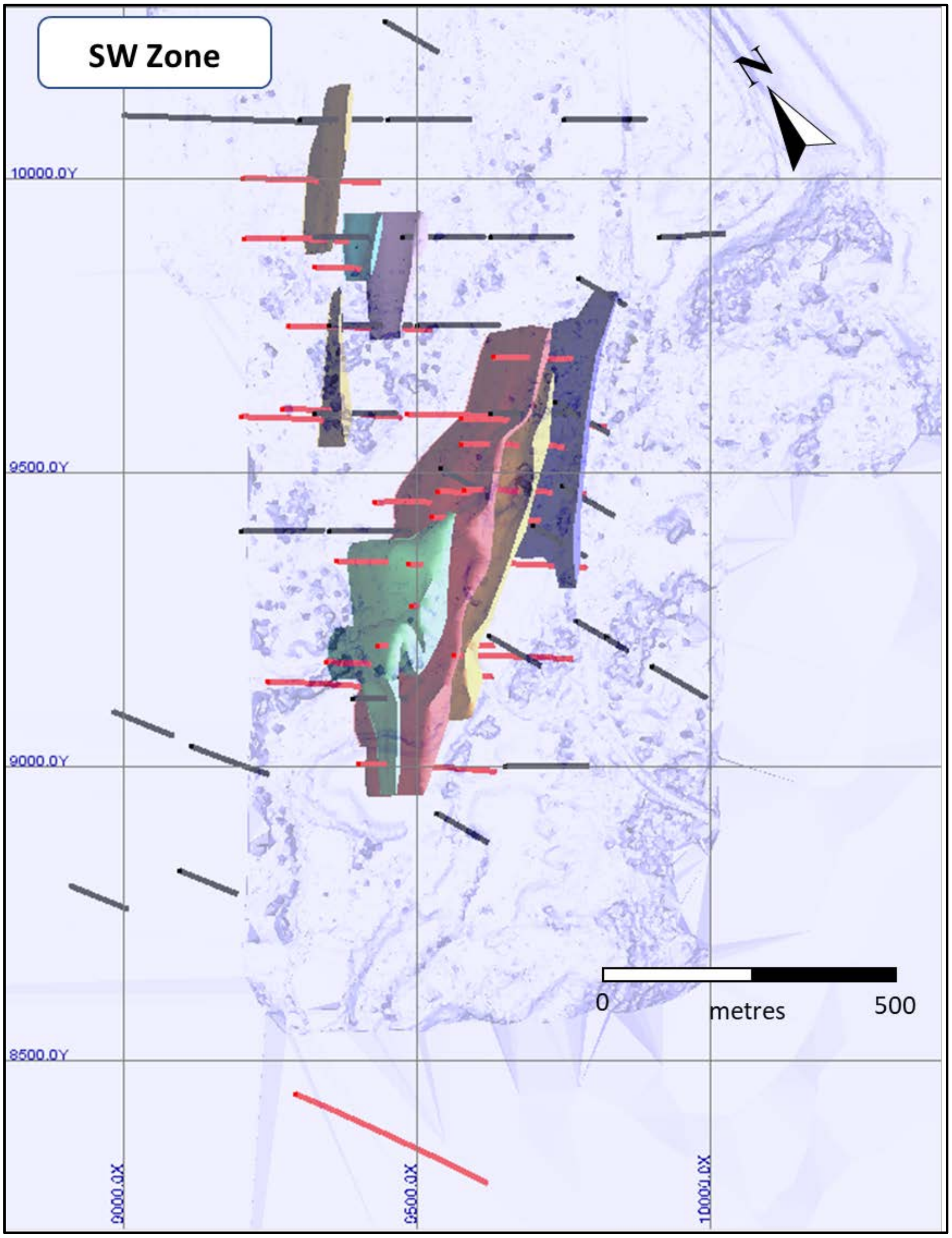
Figure 10-2 and Figure 10-3 present the drill hole locations on the Z87 and J4.J5 Zone and the SW Zone, respectively. Black drill hole traces are pre-2018, blue traces are from 2018, and red traces are from 2019 and 2020.

Figure 10-2: Drill Hole Location Map – Z87 and J4/J5 Zones



Source: AGP (2020)

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



Source: AGP (2020)

10.2 Troilus, Drill Methods and Logging, 2018 - 2020

Troilus completed its own drilling on the Property between 2018 and 2020. Troilus contracted Chibougamau Diamond Drilling Ltd. (Forages Chibougamau Ltée), based in Chibougamau, Quebec. 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 holes 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.

Drill holes were surveyed downhole using either a Reflex or EZ Gyro device. A Multishot survey was carried out from the end of each hole (Reflex by 3 m increments: EZ GYRO by 20 m increments). Drill holes were initially located in the field using either a differential global positioning system (GPS) or a handheld GPS.

10.2.1 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)

10.2.2 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 bags. 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 pre-fabricated crates (on palettes) and is covered with a plywood cover and 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, pre-2018

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 Summary of Drill Intercepts

10.4.1 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.

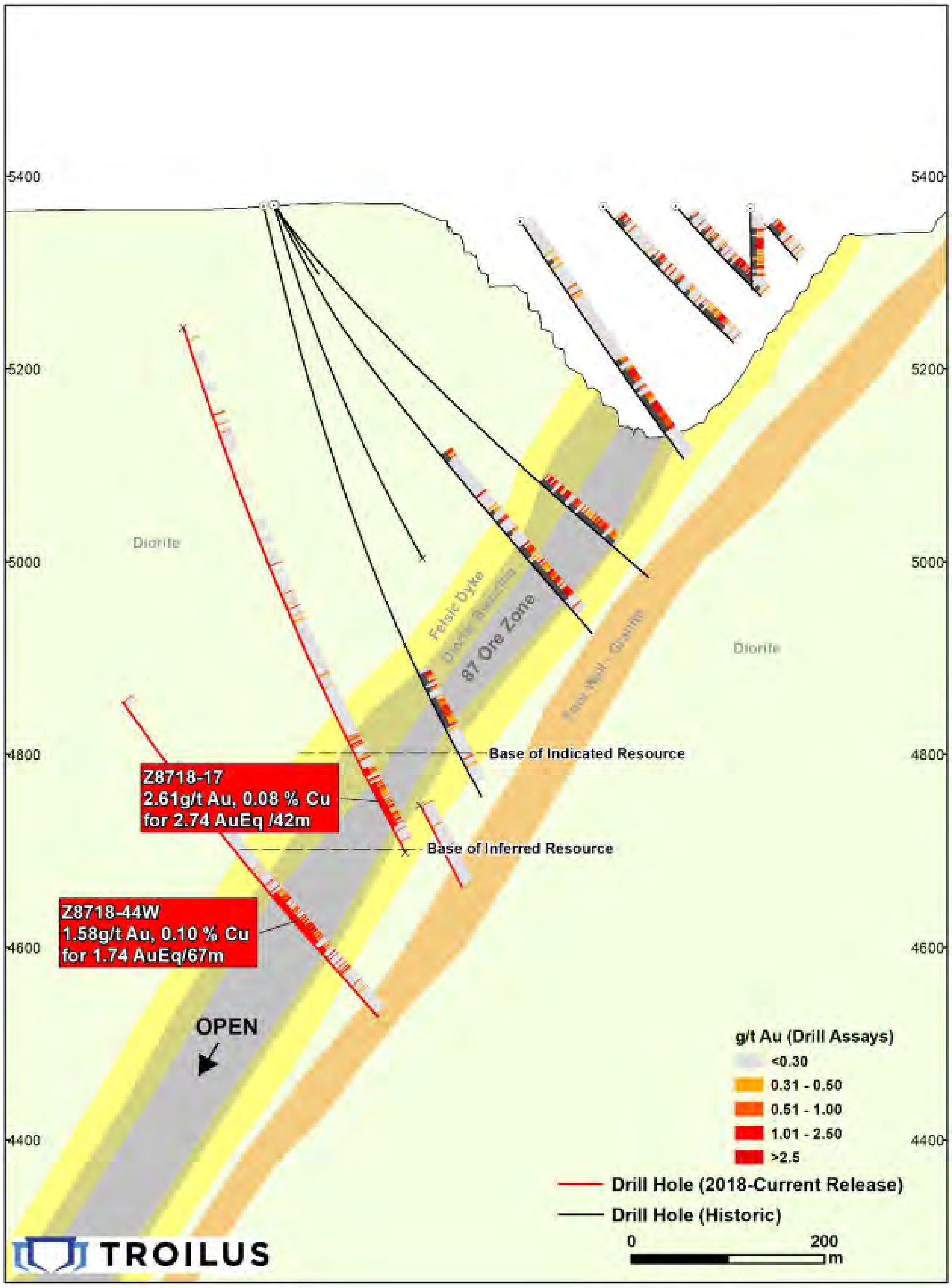
Table 10-2 lists selected drill hole intercepts with significant values. Figure 10-6 shows a cross-section of the 87 Zone at 13925 N.

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-Z8718-002	13400N		476	524	48	1.49	0.14
		including	487	491	4	7.33	0.47
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-Z8718-007	13875N		432	520	88	0.93	0.08
		including	476	477	1	31.27	0.02
		including	503	507	4	3.55	0.49
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
		including	239	241	2	1.80	0.27

Troilus Press releases: 24 May 2018; 9 Jul 2018; 12 Sep 2018; 31 Oct 2018; 19 Aug 2019 (most recently viewed 12 Jun 2020

Figure 10-6: Cross Section 13925N – Z87 Zone; looking southwest



Source: Troilus Press Release 31 Oct 2018

10.4.2 J4/J5 Zone

In 2019, the drill program focussed on the extension of the mineralization at J4/J5 zone. The drill results confirmed that the mineralization is in agreement with previous drill campaigns. Troilus drill holes have also shown that mineralization continues to the north and to the south of the J4/J5 Zone and at depth.

Table 10-3 lists selected drill hole intercepts in the J4/J5 Zone with significant values. Figure 10-7 shows a selected cross-section of the J4/J5 Zone at 14150N.

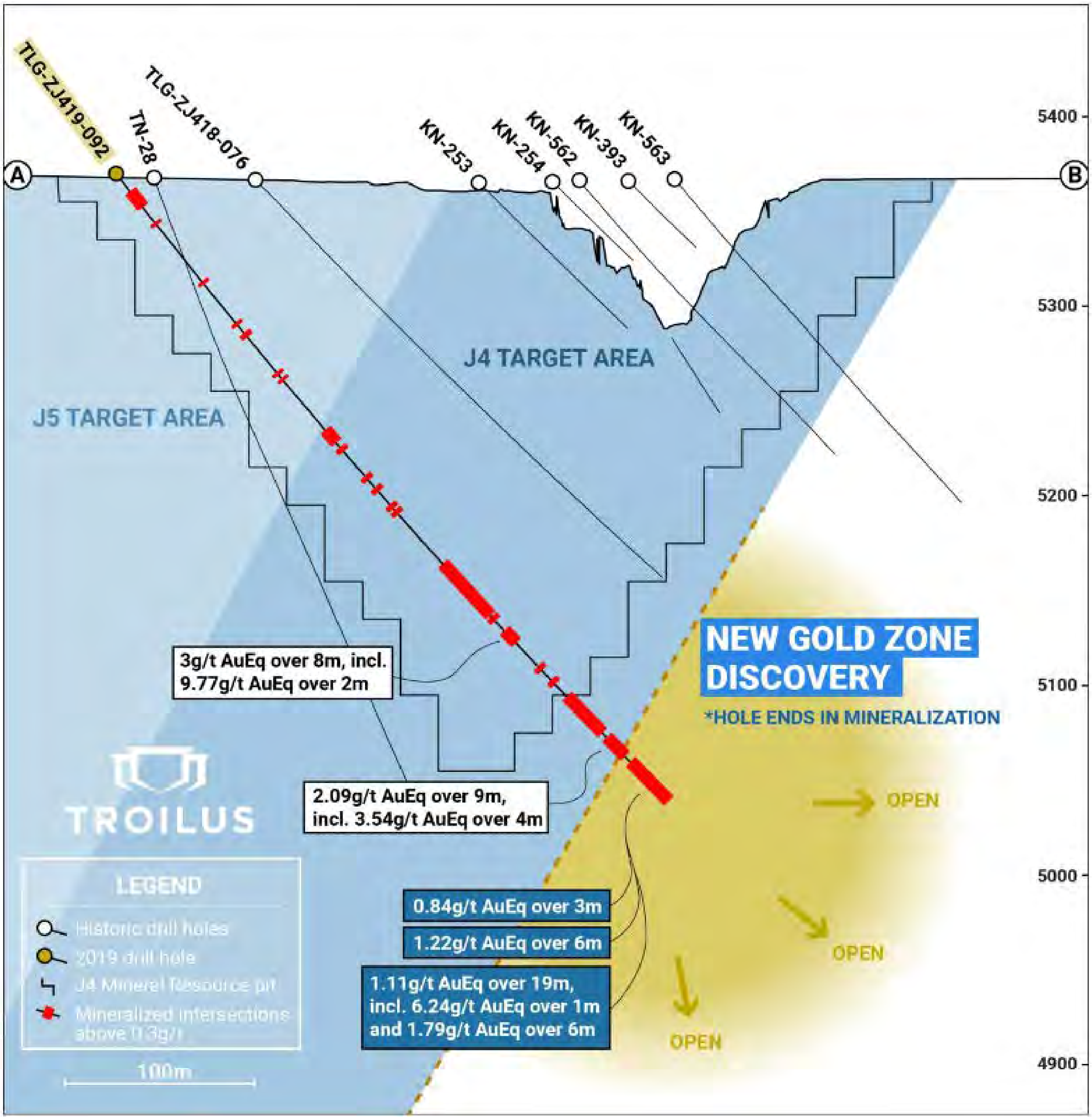
Table 10-3: Summary of Significant Drill Intercepts – J4/J5 Zone

DH No	Section		From (m)	To (m)	Width (m)	Au (gpt)	Cu (%)
TLG-ZJ418-064	14825N		258	262	4.00	1.30	0.07
			265	268	2.80	1.80	0.03
			272	283	10.72	2.20	0.05
		including	275	277	1.89	3.50	0.05
		including	282	283	0.72	7.50	0.03
			294	298	4.00	0.90	0.02
TLG-ZJ418-065	14750N		135	138	3.00	1.24	0.02
			244	247	2.60	1.00	0.07
			264	283	19.00	3.50	0.06
		including	276	282	5.65	9.40	0.07
			288	291	3.00	1.20	0.03
			303	312	9.00	0.80	0.04
TLG-ZJ418-066	14700N		136	139	3.00	2.10	0.06
			268	277	9.38	1.30	0.09
		including	274	277	3.38	2.30	0.09
TLG-ZJ418-067	14650N		246	257	11.00	0.80	0.07
			265	269	4.00	0.80	0.05
			288	292	4.00	0.50	0.08
TLG-ZJ418-068	14600N		137	139	1.59	1.50	0.07
			148	151	3.00	0.90	0.04
			218	221	3.00	3.10	0.07
		including	220	221	1.00	7.40	0.08
			225	227	2.00	1.30	0.05
			252	261	9.00	1.00	0.05
		including	257	261	4.00	1.60	0.05
			287	290	3.50	0.70	0.06
TLG-ZJ418-070	14500N		142	145	2.78	4.60	0.09
		including	143	145	1.78	6.60	0.10
			222	236	14.00	0.90	0.14

DH No	Section		From (m)	To (m)	Width (m)	Au (gpt)	Cu (%)
TLG-ZJ418-071	14400N		197	204	7.00	0.90	0.08
			207	210	3.00	0.90	0.04
			215	218	3.00	3.20	0.05
		including	215	216	1.00	7.70	0.04
			220	232	12.00	0.70	0.07
			236	245	9.00	1.00	0.08
		including	243	245	2.00	2.20	0.08
			249	256	7.00	1.00	0.09
			274	283	9.00	0.80	0.07
TLG-ZJ418-072	14350N		204	219	15.00	1.80	0.10
		including	205	207	2.00	8.30	0.15
			234	243	9.00	1.00	0.04
			246	258	12.00	1.20	0.05
		including	247	250	3.00	1.70	0.07
			266	268	2.00	1.60	0.70
			271	275	4.00	1.10	0.04
			279	291	12.00	1.00	0.07
		including	288	291	3.00	1.60	0.04
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

Source: Troilus Press releases: 14 Nov 2018; 26 Mar 2019; (most recently viewed 12 Jun 2020)

Figure 10-7: Cross Section 14150N – J4/J5 Zone; looking north



Source: Troilus Press Release 31 Oct 2018

10.4.3 SW Zone

The SW Zone is situated approximately 3.5 km southwest of the Z87 Zone pit, or 3.5 km south on the mine grid. This zone covers an area approximately 1,200 m x 500 m where Troilus completed 24 drill holes. Of this drilling, 23 of the Troilus drill holes intersected the SW Zone.

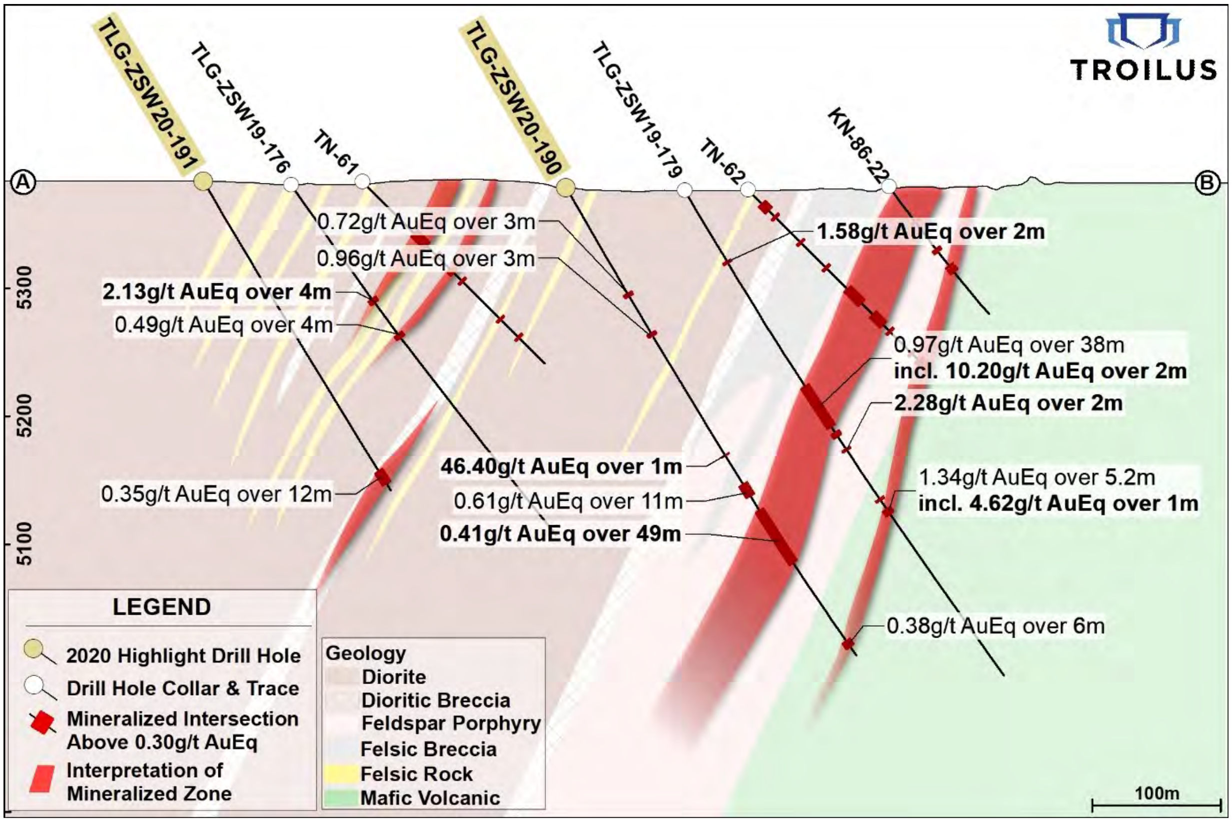
Table 10-4 lists selected drill hole intercepts in the SW Zone with significant values. The results demonstrate the presence of mineralization and, together with historical drilling, has sufficient information to complete a preliminary resource estimate. Figure 10-8 shows a selected cross-section of the SW Zone at 9600 N.

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

DH No	Section		From (m)	To (m)	Width (m)	Au (gpt)	Cu (%)
TLG-ZSW19-173	9150N		232.3	264	31.7	1.00	0.01
		including	253	260	7	2.26	0.03
			269	277	8	1.10	0.03
		including	272	275	3	2.36	0.03
			317	321	4	0.47	0.35
TLG-ZSW19-174	9200N		8.5	14	5.5	0.74	0.02
		including	10	11	1	2.31	0.06
			31	43	12	0.83	0.01
		including	31	33	2	2.16	0.00
		including	40	41	1	2.06	0.02
			63	81	18	0.70	0.04
		including	63	65	2	1.28	0.04
		including	77	81	4	1.41	0.10
			168	171	3	1.06	0.01
TLG-ZSW19-175	9350N		99	116	17	1.32	0.06
		including	99	101	2	1.94	0.03
		including	104	105	1	6.73	0.09
		including	112	114	2	3.30	0.24
			124	126	2	0.67	0.08
			134	143	9	0.43	0.01
			168	173	5	2.46	0.03
		including	172	173	1	10.60	0.11
TLG-ZSW20-185	9450N		52	6	14	0.85	0.09
TLG-ZSW20-189	9450N		14	20	6	0.80	0.24
			158	171.2	13.2	0.75	0.13
		including	158	161	3.	1.41	0.27
		including	169	171.2	2.2	1.81	0.19
			193.15	266	72.85	1.27	0.14
			210	258	48	1.7	0.17
TLG-ZSW20-190	9600N		95	98	3	0.63	0.05
			131	134	3	0.85	0.05
			242	243	1	46.30	0.04
			268	280	11	0.48	0.07

Source: Troilus Press releases: 28 Jan 2020; 21 Apr 2020; 14 May 2020 (most recently viewed 12 Jun 2020)

Figure 10-8: Cross Section 9600 N – SW Zone; looking northeast



Source: Troilus (2020)

10.4.4 J4N Zone (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 J4N (or Allongé) Zone along three fences. This zone is situated approximately 350 m to 1400 m northeast of the J4/J5 Zone. Six of the drill holes had intersections, between 2m and 12 m, of greater 0.3 gpt Au. The most significant intersections found in the Troilus drilling, approximately 900 m northeast of the J4/J5 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.

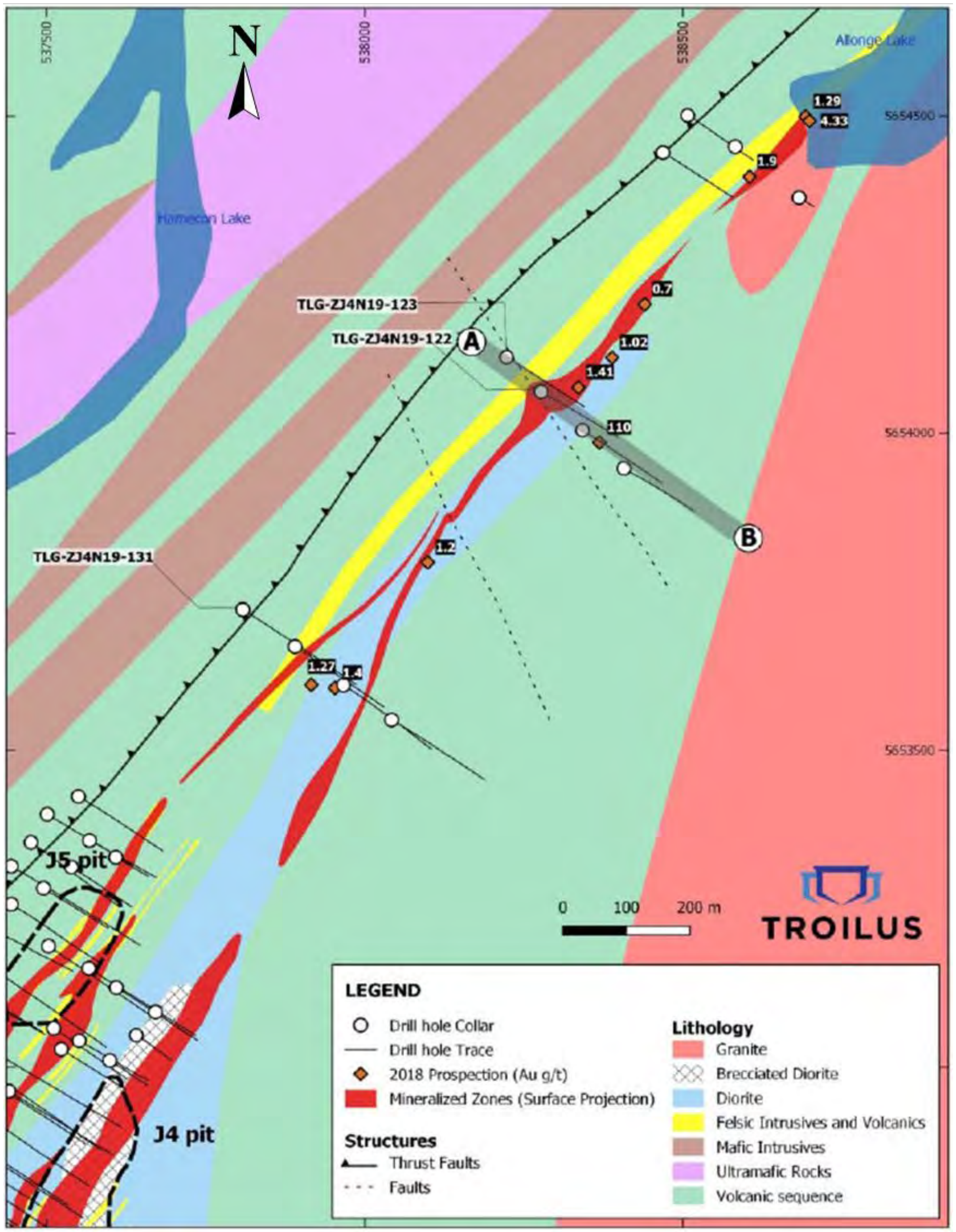
Table 10-5 summarizes the significant intersections in J4N Zone. Figure 10-9 shows the location of the J4N Zone drilling.

Table 10-5: Summary of Significant Drill Intercepts – J4N or 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

Source: Troilus Database (2020)

Figure 10-9: Plan View – J4N or Allongé Zone



Source: Troilus (2019)

10.5 AGP Opinion

AGP believes 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 Troilus, 2018 - 2020

11.1.1 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, that are listed on the SCC website (www.scc.ca).

11.1.2 Sampling 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 jsAnalytical 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 acid 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.

11.2 Pre-2018

11.2.1 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.

11.2.2 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.

11.3 Density Determinations

11.3.1 Z87 Zone

Historically, density measurements from 2,721 core samples in the 30 deep drill holes (KN-648 to KN-677) were collected by Inmet (RPA, 2006). The core samples tested were generally whole core pieces ranging in length from approximately 10 cm to 20 cm. Mine personnel weighed samples in air and in water, and the density results were adjusted to account for water temperature. Measurements on 496 resource related samples range from 2.57 g/cm³ to 3.42 g/cm³ and average 2.86 g/cm³. The same average is obtained when the lowest ten and highest ten density measurements are excluded (RPA, 2019a). These measurements are used for the current resource estimate.

The historic density measurements were unavailable at the time of writing. AGP recommends a review of this data when the site is re-opened.

Z87 South, 2019

During the 2019 drill program on Z87S Zone, Troilus began their density measurements core samples in the Z87S 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 4,255 measurements were collected from 22 drill holes; 526 of these values intersect eight domains. These density records were received late and were not included in the current resources. AGP reviewed the data and location of the measurements and determined that they represent less than 1% of the total Z87 Zone resources.

Table 11-1: Descriptive Statistics for Density – Z87 Zone

	Domain Density (g/cm^3)	Country Rock Density (g/cm^3)
Count	526	3,729
Minimum	2.46	2.07
Maximum	3.18	3.23
Mean	2.75	2.77
Median	0.09	0.11
StDev	0.01	0.01
CV	0.03	0.04

AGP recommends a review of all available density data in the Z87 and Z87S areas be carried out collectively to determine the best possible representation of density for this Zone. AGP also recommends the collection of density data from recent drilling in the Z87 Zone as well.

11.3.2 J4/J5 Zones

During the drill program on the J4/J5 Zone, Troilus has density measurements completed on core samples. Density measurements were carried out by ALS (Sudbury) (ALS Code: OA-GRA08) on selected drill core samples set for assay analysis using water immersion (wet/dry) method. A total of 13,409 measurements were collected from 46 drill holes; 3,356 of these values intersect all 13 of the interpreted mineralized domains. For the current resource estimate a single mean density is used for the J4/J5 Zone. Table 11-2 presents the statistics for density in the J4/J5 Zone.

Table 11-2: Descriptive Statistics for Density by Mineralized Domain – J4/J5 Zone

	Domain Density (g/cm^3)	Country Rock Density (g/cm^3)
Count	3356	10053
Minimum	2.20	2.10
Maximum	3.20	3.63
Mean	2.80	2.77
Median	2.80	2.77
StDev	0.10	0.07
CV	0.04	0.03

AGP recommends a review of the spatial distribution of the density data in the J4/J5 Zone to determine if density may be estimated or assigned by mineralized domain.

11.3.3 SW Zone, 2019 – 2020

During the 2019-2020 drilling campaign, Troilus collected density readings collected for all sample intervals.

During the drill program on the SW Zone, density measurements carried out by ALS (Sudbury) on drill core samples sent for assay analysis using water immersion (wet/dry) method. A total of 8,524 measurements were collected from all 24 drill holes; 1,222 of these values intersect all 8 of the interpreted mineralized domains.

For the current resource estimate density was assigned by mineralized domain for the SW Zone. AGP recommends continued collection of density measurements to further characterize both mineralized domains and country rock. A review of the spatial distribution of the density data in the SW Zone can then be carried out to determine if density may be estimated or assigned by mineralized domain.

Table 11-3 presents the statistics for density in the SW Zone, by mineralized domain.

Table 11-3: Descriptive Statistics for Density (g/cm^3) by Mineralized Domain – SW Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
201	321	2.64	3.85	2.81	2.80	0.10	0.04
202	377	2.42	2.97	2.72	2.70	0.07	0.03
203	70	2.66	2.83	2.72	2.71	0.04	0.01
204	149	2.08	3.07	2.91	2.92	0.11	0.04
205	199	2.61	2.98	2.80	2.80	0.05	0.02
206	8	2.68	2.93	2.84	2.85	0.09	0.03
207	25	2.56	2.81	2.75	2.77	0.06	0.02
208	73	2.66	2.88	2.79	2.80	0.06	0.02
Country Rock	7302	1.81	3.71	2.81	2.79	0.11	0.04

11.4 Quality Assurance / Quality Control

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)
- standard (SRM) (1 in every 25 samples)

Analytical QAQC 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.

Standards

Troilus used five commercially produced Certified (or Standard) Reference Materials (CRMs) during the drill programs from Ore Research & Exploration PL, based in Perth, Australia.

These CRMs are summarized with their 'recommended values' in Table 11-4 below.

Table 11-4: Standard Reference Materials (SRMs) and Recommended Values

Troilus Number	SRM	Source	Au (gpt)	Cu (ppm)	Ag (gpt)
S1	OREAS 209	Ore Research & Exploration PL	1.58	76	0.264
S2	OREAS 215		3.54	-	-
S3	OREAS 217		0.338	-	-
S4	OREAS 92		-	2294	0.70
S5	OREAS 922		-	2122	0.888

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 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 in through out all drilling programs. Due to the variable nature of gold within the sample pulps and rejects, duplicate samples were deemed too inconsistent to be of use and stopped in July 2019.

11.4.1 QA/QC, 2018 – 2019

The QA.QC program included blank materials and CRMs. Four CRM's 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 11-5 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 11-6 shows a summary of the QA/QC samples submitted during the 2019-2020 drilling program on the SW Zone.

Table 11-5: Summary of Troilus QAQC 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.

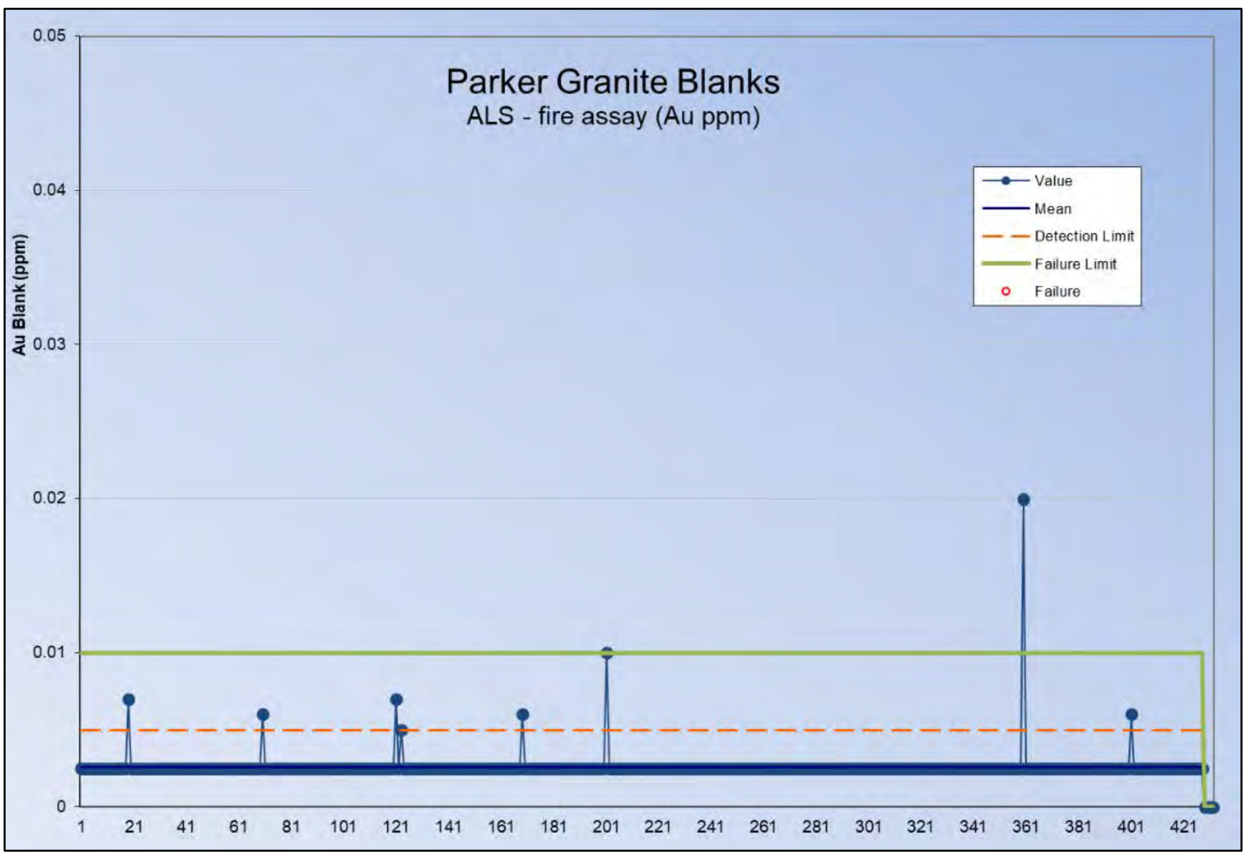
Table 11-6 shows the results of the blanks used in the 2018 – 2019 drilling. Figure 11-1 and Figure 11-2 present plots for fire assay blanks and metallic sieve assay blanks, respectively.

Table 11-6: 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	-

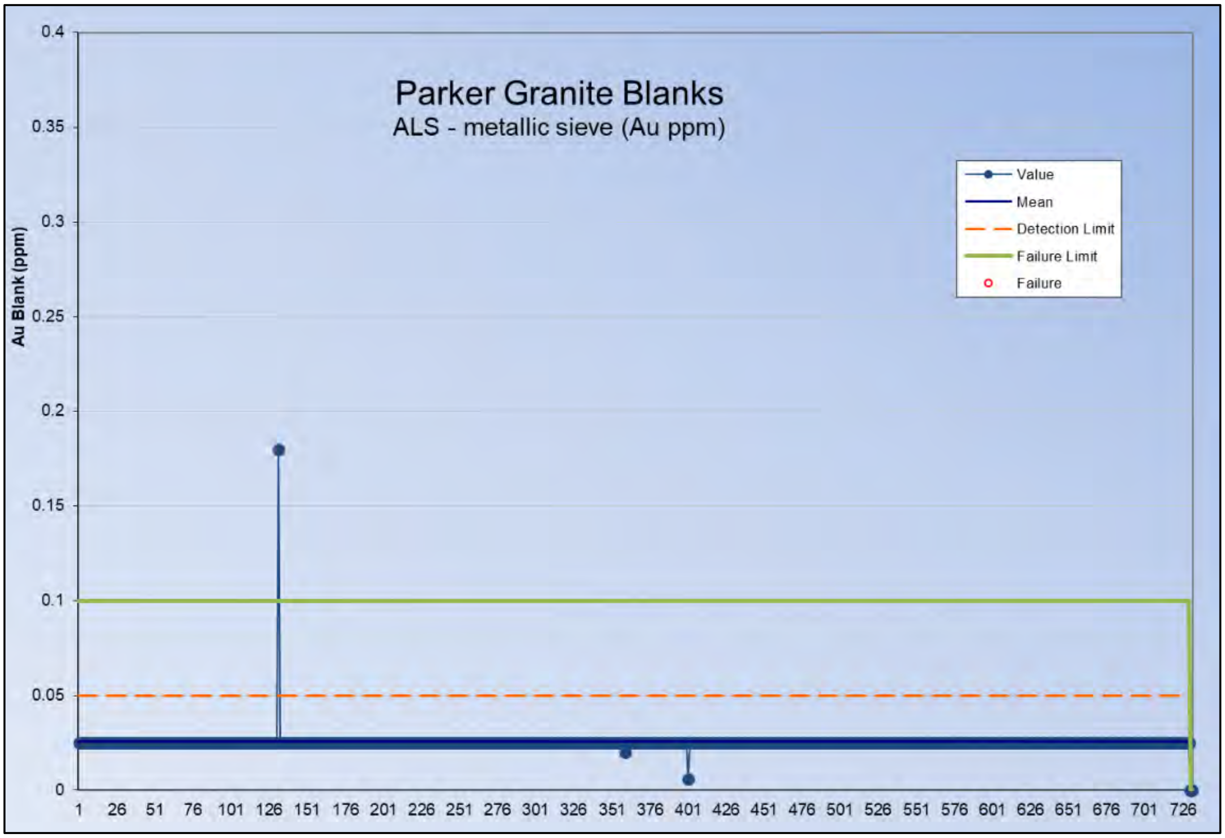
BP -Parker Granite Coarse Blank

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



Source: AGP (2020)

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



Source: AGP (2020)

Standards

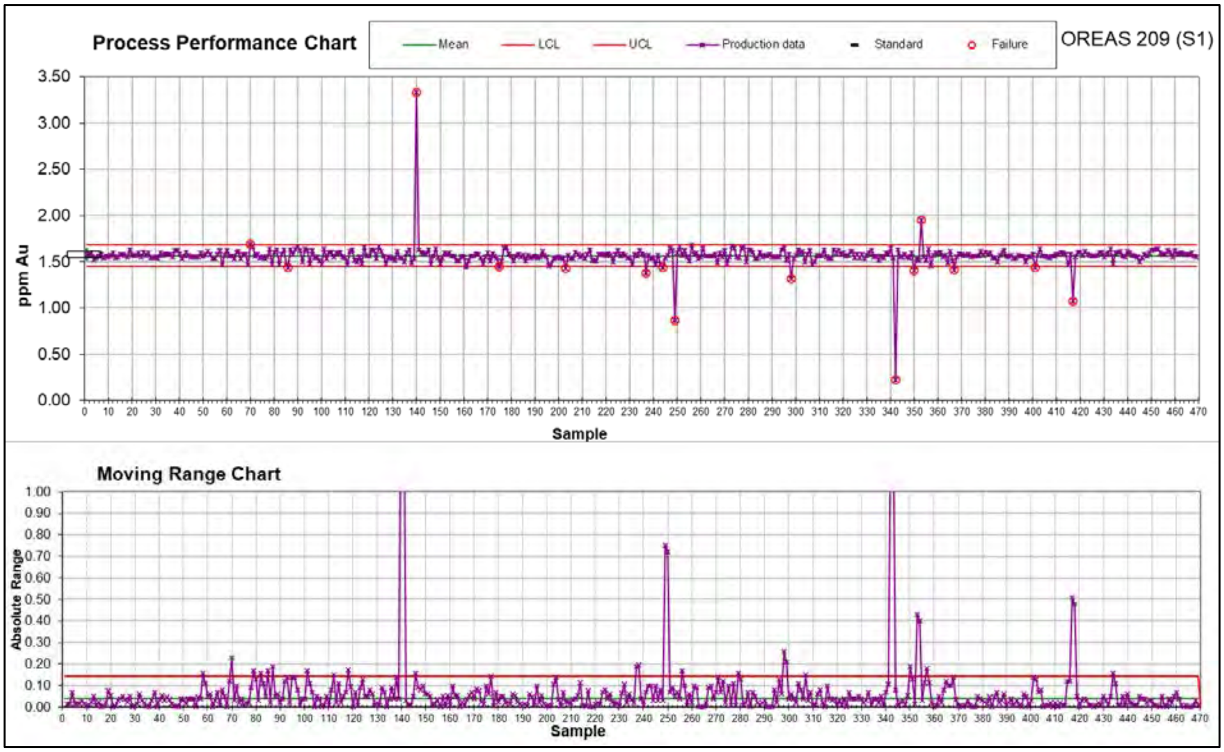
Table 11-7 presents the results of the CRMs used in the 2018-2019 drilling. Figure 11-3 presents accuracy plot for gold from the 2018 and 2019 drilling.

Table 11-7: 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%

BP -Parker Granite Coarse Blank

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



Source: AGP (2020)

11.4.2 QA/QC, 2019 – 2020 (SW Zone)

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 CRM's. Table 11-8 shows a summary of the QA/QC samples submitted during the drilling program.

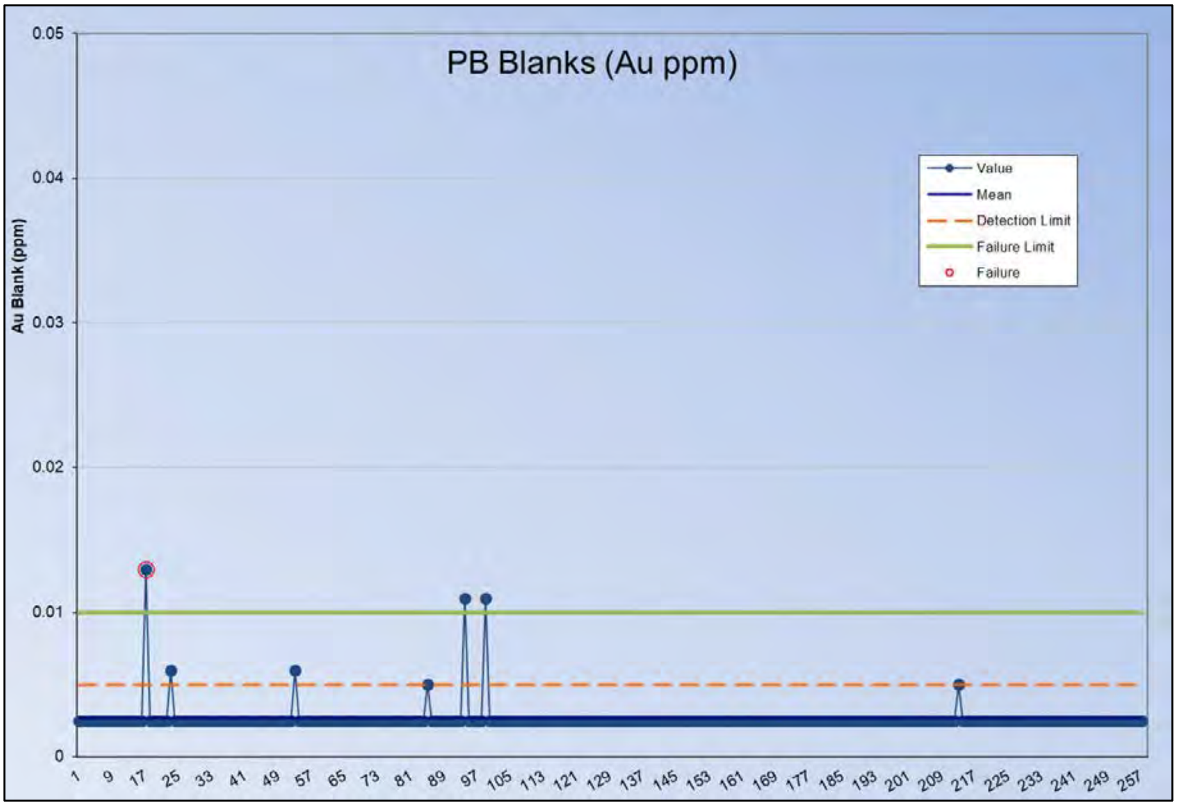
Table 11-8: Summary of Troilus QAQC Program, 2019 – 2020

Description	Number of Samples (% of database)
Total Number of Samples	8,525
Number of Control Samples	743 (8.7%)
Distribution	
Blanks (BP)	376 (4.4%)
CRM samples	376 (4.3%)
OREAS 209 (S1)	99
OREAS 215 (S2)	110
OREAS 217 (S3)	90
OREAS 922 (S5)	68

Blanks

During the 2019 – 2020 drilling on the SW Zone, only 3 failures occurred. These were verified and all were less than 0.014 ppm Au and determined not to have a significant impact on the sample batches. Figure 11-4 presents the plots for the gold assay blanks.

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



Source: AGP (2020)

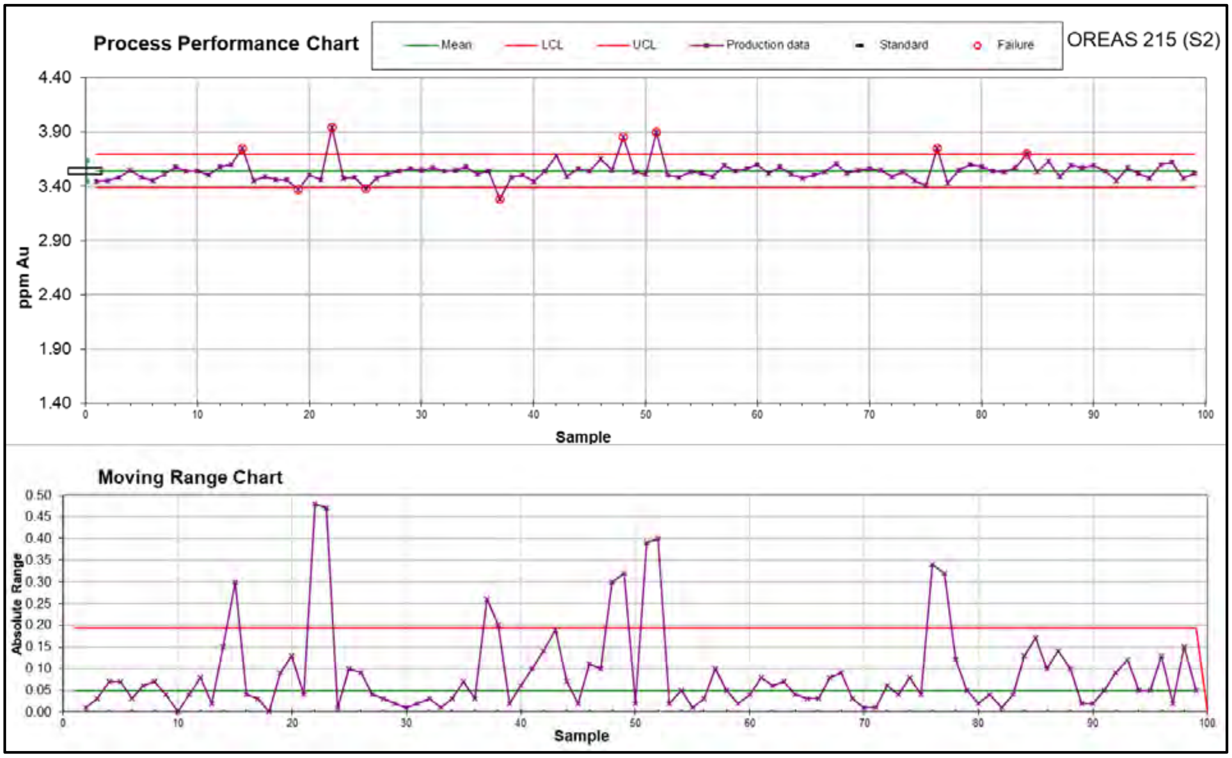
Standards

Table 11-9 presents the results of the CRMs used in the 2019-2020 drilling. Figure 11-5 presents the accuracy plot for gold for the OREAS 215 CRM.

Table 11-9: CRM Results, 2018 – 2019 Drilling

CRM	Recommended Value	Standard Deviation	Number of Samples	Number of Failures	Percent Failure
OREAS 209 (S1) ppm Au	1.58	0.044	55	0	-
OREAS 215 (S2) ppm Au	3.54	0.097	99	9	9.1%
OREAS 217 (S3) ppm Au	0.338	0.010	87	2	2.3%
OREAS 922 (S5) %Cu	0.212	0.009	64	3	4.7%
OREAS 922 (S5) ppm Ag	0.888	0.109	64	2	3.1%

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



Source: AGP (2020)

11.4.3 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.

11.5 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.

11.7 AGP Opinion

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.

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 reasonable. AGP recommends that the initial drilling at Z87 Zone be submitted to ALS for density measurements and that a review of the Inmet and Troilus density data be undertaken. AGP recommends that density measurements continue to be collected in future drill programs.

12 DATA VERIFICATION

12.1 Data Verification

12.1.1 Z87 Zone, J4/J5 Zone

AGP received the database for all drill holes in the J4/J5, Z87 and SW Zones as a Geovia GEMS project database. AGP also received the exported CSV files of the Troilus drill holes from the Geotic database.

the database for the J4/J5 Zone and the Z87 Zone as a Geovia GEMS project database; and CSV files, exported from Troilus' Geotic database.

AGP verified the database for the J4/J5 Zone and the Z87 Zone using GEMS validation tool to determine whether there were missing overlapping intervals. The drill holes were also checked visually for any misplaced drill hole collars. No errors were found. AGP verified approximately 5% of the J4/J5 and Z87 Zone drill holes comparing the gold, copper, and silver assays to the laboratory certificates. No errors were found.

12.1.2 SW Zone

AGP received the database for the SW Zone as a Geovia GEMS project database; and as csv files, exported from Troilus' Geotic database. AGP verified the database using the GEMS validation tool to determine whether there were missing and/or overlapping intervals. The drill holes were also checked visually for any misplaced drill hole collars. No errors were found.

For the Troilus drill holes, the assay values in the database were compared against the assay certificates provided to Troilus by ALS. AGP verified approximately 14% of Troilus' assay values and no errors were found.

AGP verified four historic drill logs in the SW Zone to review on site to compare drill collar and assay values in the database and no errors were found. AGP also visually checked the historic drill hole in the GEMS database and found no issues.

12.2 AGP Site Visit

The site visit to the Project was conducted by the QP from 18 February to 20 February for two days. The 2020 drill program was in progress and near completion during the site visit. The author was accompanied on the site visit by M. Thiago Diniz, Technical Manager for Troilus and M. Bertrand Brossard, Senior Project Geologist for Troilus. The site visit included an inspection of core logging, sampling, and core storage facilities, checking of drill hole collar coordinates, and reviewing drill core logs against selected drill core.

12.2.1 Drill Core Logging and Sampling and Storage Facilities

Drill core for the Project is logged, sampled, and stored in the rear of a permanent warehouse on the mine site (Figure 12-1) where the front serves as a garage. This facility also serves as an office work stations for Troilus exploration personnel. A semi-permanent exploration office is situated next to the warehouse (Figure 12-2). Outside the warehouse, Troilus has constructed covered drill core racks for the storage of drill core.

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

Figure 12-1 shows the three warehouses used for core logging and core storage. Figure 12-2 shows the interior of the core logging facility.

Figure 12-1: Drill Core Logging and Sampling Facility

Source: AGP (2020)

Figure 12-2: Exploration Office

Source: AGP (2020)

12.2.2 Drill Hole Collar Locations

AGP located 13 drill hole collars at the SW Zone, 87 Zone and J Zone. Due to severe cold and wind and snow coverage at the time of the site visit, additional drill holes were not spotted.

The locations of diamond 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. The drill hole is marked by a metal rod topped by a red- painted metal rod and, in some cases with a wood stake (marking the planned drill hole) can be seen above the level of snow (Figure 12-3).

Figure 12-3: Drill Hole Collars for TLG ZSW20-191 (SW Zone) and TLG Z87S19-146 (Z87 Zone)

Source: AGP (2020)

The collar coordinates measured by AGP fell within a 4 m tolerance of those reported by Troilus. It is the QP's 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 for checked drill holes.

Table 12-1: Comparison of Collar Location Coordinates at the Troilus Project; NAD83 Zone 18U

Drill Holes	Troilus Easting (m UTM)	Troilus Easting (m UTM)	AGP Easting (m UTM)	AGP Easting (m UTM)	Δ Easting (m)	Δ Northing (m)
SW Zone						
TLG-ZSW19-173	534,232	5,647,957	534,234	5,647,957	-2	0
TLG-ZSW19-174	534,423	5,647,852	534,419	5,647,854	4	-2
TLG-ZSW19-176	534,397	5,648,362	534,397	5,648,363	0	-1
TLG-ZSW20-181	534,322	5,647,934	534,320	5,647,936	2	-2
TLG-ZSW20-182	534,129	5,647,982	534,129	5,647,985	0	-3
TLG-ZSW20-183	534,405	5,647,961	534,402	5,647,964	3	-3
TLG-ZSW20-191	534,333	5,648,387	534,330	5,648,388	3	-1
TLG-ZSW20-192	534,481	5,648,476	534,483	5,648,477	-2	-1
Z87 Zone						
TLG-Z8718-004	536,931	5,651,372	536,932	5,651,373	-1	-1
TLG-Z8718-007	537,183	5,651,645	537,183	5,651,646	0	-1
TLG-Z87S19-139	536,999	5,650,838	536,999	5,650,841	0	-3
TLG-Z87S19-140	536,935	5,650,879	536,933	5,650,879	2	0
J Zone						
TLG-ZJ418-077	537,038	5,651,946	537,034	5,651,949	4	-3

12.2.3 Drill Core Log Review

The site visits also included a review of the drill core logs and comparison to selected drill core intervals. The lithology descriptions and sample intervals in the drill logs were consistent with the drill core intervals reviewed. Table 12-2 lists the selected drill core intervals examined during the site visit.

Table 12-2: Selected Drill Core Intervals Examined

Deposit	Drill Hole	From (m)	To (m)
SW Zone			
	TLG-ZSW19-173	249.69	288.35
	TLG-ZSW19-175	97.59	127.70
	TLG-ZSW19-177	119.75	145.58
	TLG-ZSW19-179	183.26	213.60
	TLG-ZSW20-184	100.44	109.05
J Zone			
	TLG-ZJ518-029	159.95	177.26
	TLG-ZJ518-030	157.72	179.24
Z87 Zone			
	TLG-Z8719-119	349.69	375.12
	TLG-Z8719-132	319.92	358.55
	TLD-Z8719-134	305.55	335.69

12.2.4 Independent Samples

The collection of independent samples is meant to demonstrate that mineralization exists on the property in similar ranges as reported by the issuer. These samples are not intended to act as duplicate samples. AGP collected three samples selected from the available drill core during the site visit.

The sample intervals were selected from the 2019 drilling on the SW Zone. The samples were collected from the same sample intervals as those of Troilus for a direct comparison.

AGP supervised the quartering of the selected samples by rock saw and placed each sample in a marked sample bag, sealed with a zip tie. A sample tag was stapled in the core box at the location of the AGP sample. Collected samples were transported by AGP to Toronto and couriered to Activation Laboratories Ltd. (ActLabs) in Ancaster, Ontario for assay analysis.

Once received at ActLabs, samples were prepared by crushing the sample to 80% passing 10 mesh and then a split of 250 g was pulverized to 85% passing 200 mesh (ActLabs code: RX1). Samples were analyzed for 42 elements by four acid digestion and ICPOES/ICPMS method (ActLabs code UT-4M). Gold was analyzed separately by fire assay and atomic absorption (ActLabs Code 1A2B-30). The list of independent sample is shown in Table 12-3 and the comparison of result are presented in Table 12-4.

Table 12-3: Summary of Independent Samples

AGP Sample No.	Troilus Sample No.	Drill Hole	Core Box(es)	Sample Interval (m)
AO266259	AO265751	TLG-ZSW19-173	57	261 – 262
AO269424	AO265752	TLG-ZSW19-177	27	131 – 132

Table 12-4: Independent Sample Results

Sample No.	Drillhole	Au (gpt)	Ag (gpt)	Cu (%)
AGP				
	TLG-ZSW19-173	0.95	0.40	0.010
	TLG-ZSW19-177	2.90	0.80	0.28
Troilus				
	TLG-ZSW19-173	0.67	0.25	0.004
	TLG-ZSW19-177	1.23	0.70	0.30
Difference				
	TLG-ZSW19-173	-0.29	-0.15	-0.01
	TLG-ZSW19-177	-1.67	-0.10	0.02

AGP considered the grade range of the representative samples to be acceptable and demonstrated the presence of mineralization on the Property in the same tenure as reported by Troilus. AGP interprets the differences of gold grades of the independent samples to be due to the degree of variability of the gold mineralization.

12.3 QP 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

The historical Troilus mill was commissioned in 1996, with commercial production achieved in April 1997 at a rate of 10,000 tpd. This mill was designed to treat gold, copper, and silver using a flowsheet that utilized gravimetric, froth flotation, and cyanidation circuits. Copper concentrate and doré bars were produced on site, with average recoveries of 86% Au and 90% Cu and a flotation concentrate grade of 18% Cu. By the end of 1998, the plant reached production of 10,850 tpd with similar metallurgical results.

At the beginning of 1998, a decision was made to increase mill capacity to 15,000 tpd using a coarser grind. A crushing and screening plant was constructed and became operational in early 1999. The objective was to reduce the critical size material in the feed down to less than two inches. The cyanidation portion of the flowsheet was dropped in 1999, since it was found to be uneconomic. Removing the cyanidation circuit decreased the gold recovery by 2%, while coarser grind was responsible for approximately a further 1% to 1.5% decrease. Since 1999, the plant was operated with gold recoveries in the 82.5% to 84% range.

At the end of 2001, after replacement of the pebble crusher and ball mill pump, plus the successful implementation of instrumentation upgrades and various flowsheet changes, the plant reached its target tonnage capacity. Similarly, steps were undertaken in 2000 to improve copper metallurgy, particularly concentrate grade. A column cell was commissioned, and modifications were carried out to the copper cleaner and thickening circuit. These changes led to improvements in the concentrate grade by 3% copper and recovery improvements by 1% to 2%. More importantly, this permitted the mill to operate more efficiently in a wider range of copper feed grades.

Plant recoveries in 2005 were approximately 82% for gold and 90% for copper (Figure 13-1 and Table 13-1). In 2004, the plant reached a new milestone of 18,000 tpd.

Figure 13-1: Historical Gold and Copper Mill Recoveries, Troilus Gold Mine

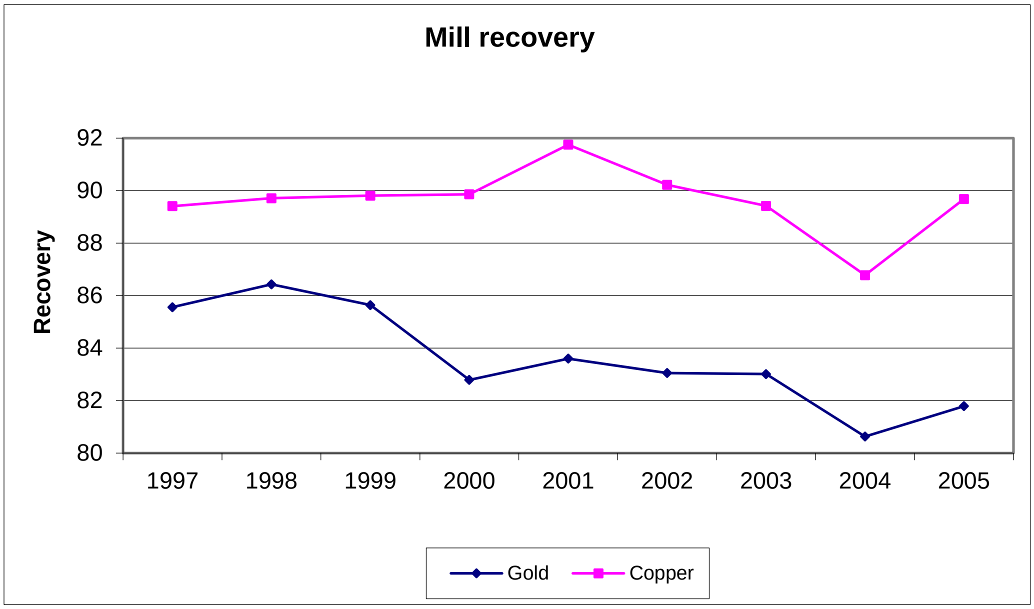


Table 13-1: 2005 Operational Results Summary

Material	Weight %	Assays		Distribution	
		(% Cu)	(gpt Au)	(% Cu)	(% Au)
Mill Feed	100	0.07	0.98	100	100
Concentrate	0.40	17.33	128.76	89.68	54.43
Gravity					27.36
Final Tailing	99.60	0.008	0.17	10.32	18.21

A number of metallurgical testwork programs have been conducted on samples of mineralization from various zones within the Troilus deposit, dating back to the early 1990s. A complete reference list is given within Section 27.

13.2 Metallurgical Testwork

13.2.1 Comminution Testing

JK Drop weight testing was conducted on four composites at COREM in June 2020 and results were interpreted by JKMRC in July 2020. The composites were identified as Domain 41, Domain 42, Domain 43, and Domain 54. A summary of the JK Drop weight parameters is provided in Table 13-2 below. According to the JKMRC database of projects, the A*b values for the four composites are low, which suggests that Troilus material is "moderately hard" and has a relatively high resistance to breakage.

Table 13-2: Summary of COREM 2020 JK Drop weight Test Results

Composite ID	A	b	A*b	SCSE (kwh/t)	SG
Domain 41	56.9	0.65	37.0	10.42	2.78
Domain 42	54.3	0.72	39.1	10.21	2.81
Domain 43	57.4	0.72	41.3	10.12	2.88
Domain 54	54.8	0.77	42.2	9.71	2.74

In addition to the above, Hazen Research conducted Bond ball work index (BBWi) and abrasion index (Ai) testing on three composites from the Troilus project: 87UG, 87S and SW. The closing screen size for the Bond ball work index tests was selected to be 150µm. The results are summarised in Table 13-3 below.

Table 13-3: Summary of Hazen Research 2020 Ai and BBWi Results

Composite ID	Ai, g	BBWi, kwh/t (metric)
88029 A 87UG	0.285	9.8
88030 A 87S	0.344	11.0
88031 A SW	0.547	14.7

The 1993 Kilborn feasibility study reported the average Bond ball work index to be 9.7kWh/t, which is on the lower end of the recent comminution testwork dataset. Actual Bond rod and Bond ball work index tests undertaken on the 1993 laboratory testwork and pilot plant composites are summarized in Table 13-4 below.

Table 13-4: Summary of Lakefield 1993 Comminution Testwork Results

Composite ID	Bond Ball Work Index (kWh/t)	Bond Rod Work Index (kWh/t)
Low 87	9.4	16.6
Medium 87	9.9	16.6
High 87	9.6	15.3
J4 Zone	8.8	15.2
Pilot Plant Comp (87 Zone)	10.3	14.8

13.2.2 Metallurgical Testwork, Lakefield, 1993

Five composites from the 87 Zone and J4 Zone were evaluated at Lakefield Research, Ontario in 1993. Four of the five composites were subjected to bench scale gravity and flotation testwork to produce a gravity concentrate and a copper-gold flotation concentrate, while the “Pilot Plant” composite comprised of high grade Zone 87 material and was used exclusively for pilot plant testwork. The pilot plant included semi autogenous and ball mill grinding, gravity concentration and copper-gold flotation (rougher flotation, regrind, three stages of copper cleaning). The head grades for the 1993 Lakefield composites are summarised in Table 13-5 below:

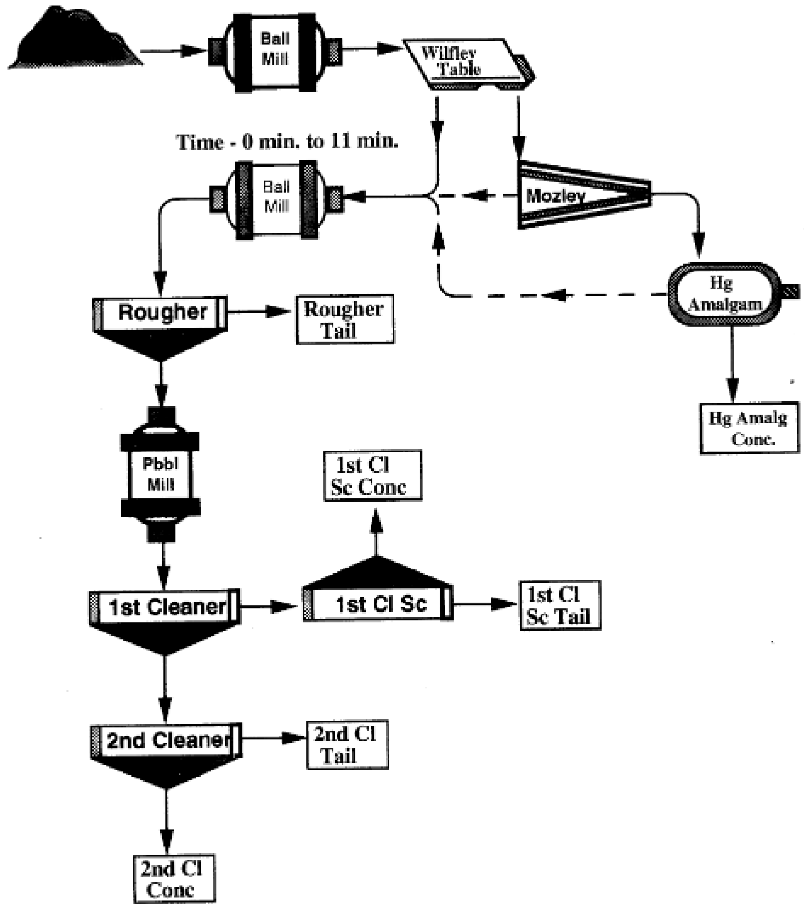
Table 13-5: Summary of Lakefield 1993 Composite Head Grades

Composite ID	Cu Head Grade, %	Au Head Grade, %
High 87 Zone	0.16	2.11
Medium 87 Zone	0.11	1.03
Low 87 Zone	0.055	0.97
J4 Zone	0.043	1.39
Pilot Plant Comp	0.19	2.06

Bench scale Laboratory Gravity & Flotation Testwork

Bench scale gravity and flotation testwork was completed on the high, medium, and low copper grade 87 Zone composites and the J4 Zone composite. The flowsheet consisted of two stage primary ball milling, intermediate gravity concentration via Mozley table, rougher flotation, regrinding and two stages of cleaner flotation with a cleaner scavenger on the cleaner 1 tails (shown in Figure 13-2 below).

Figure 13-2: General Laboratory Testwork Flowsheet Applied at Lakefield in 1993



The results of the best bench scale test for each composite are summarised in Table 13-6 below:

Table 13-6: 1993 Lakefield Bench Scale Testwork Metallurgical Performance

Composite ID	Cu Head, % Cu	Au Head, g/t Au	Cu Conc Grade, %Cu	Cu Conc Cu Rec.,%	Total (grav + float) Au Rec., %
J4 Zone	0.045	1.53	4.0	**62**	**94**
Low 87	0.058	1.06	5.0	**85**	**87**
Med. 87	0.11	0.94	10.7	**85**	**87**
High 87	0.16	1.88	17.3	**91**	**90**

At copper head grades much below 0.1% Cu, the ability to make reasonable copper concentrate grade diminished quite significantly. J4 Zone material yielded particularly low copper concentrate grade (4.0% Cu) and low copper recovery (62%), but total gold recovery was excellent at 94%. With the exception of the J4 Zone composite, both copper and gold recoveries were acceptable and averaged 87% and 88% respectively, but copper concentrate grade quality is a concern for all composites tested except for the high grade 87 Zone material. Gold gravity recovery was good for all composites tested and ranged from 37% to 56%, averaging 46%.

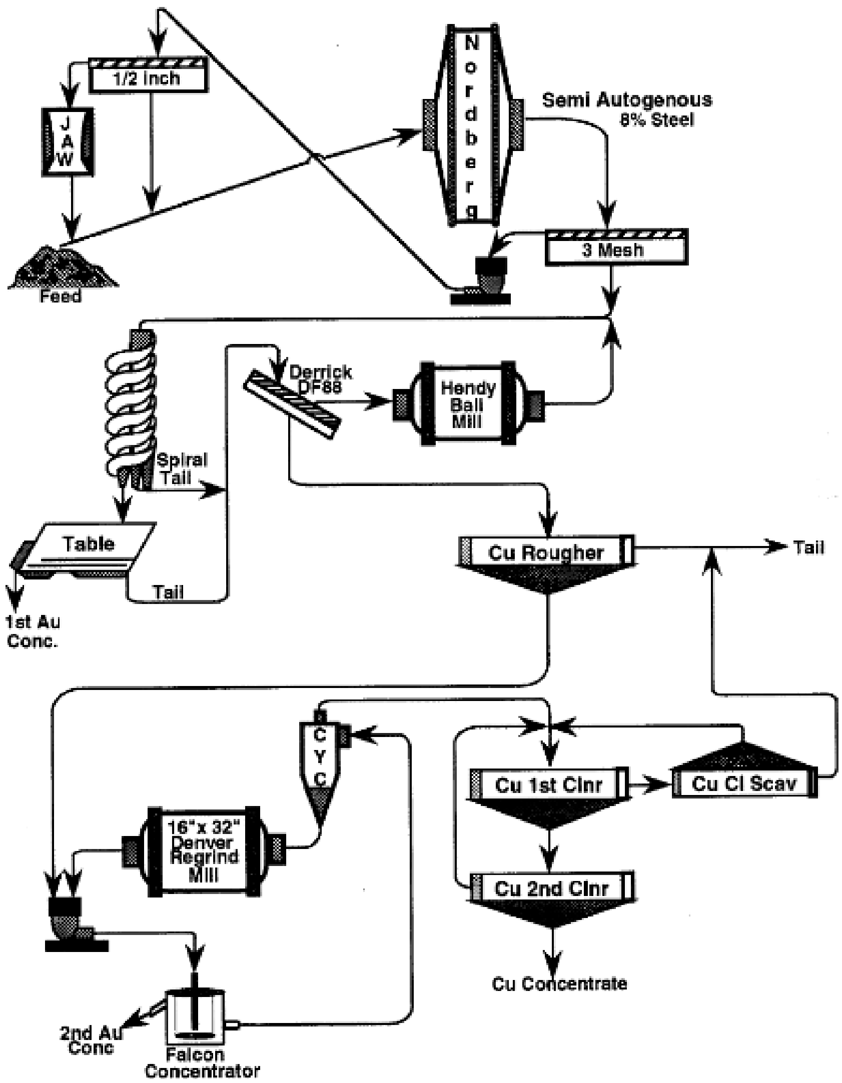
A subsequent phase of testwork was undertaken in an attempt to improve copper concentrate grades from the lower head grade composites (J4 Zone and Low 87 Zone). The addition of sodium sulphite as a pyrite depressant showed promise, raising copper concentrate grades to 10-13% Cu albeit at a lower range of copper recovery (76-78%). The use of 10-kg charges in subsequent open circuit batch gravity/flotation testing improved copper recovery to 81-85% at copper concentrate grades of 14-16%, although the locked cycle test that followed these open circuit tests however was unable to replicate these results with grades of 7% Cu and 78% copper recovery.

The final phase of testwork at Lakefield in 1994 gave rise to a significant breakthrough in copper metallurgy. Sodium sulphite (Na_2SO_3) and lime were employed in the regrind/cleaner circuit after a bulk sulphide rougher flotation step. It is again noteworthy that the tests were conducted using 10-kg charges, and it is considered likely that the increased copper units available for flotation and higher cleaner pulp densities played a significant role in improving concentrate grades and recoveries for this program. In test No. 242, the “Low 87” composite (head grade of 0.06% Cu) obtained a copper concentrate grade of 16.9% Cu at 85% copper recovery – a significant improvement over previous results on this composite. Gold assays were not conducted on this test, however. This same flowsheet was also tested on the “Zone 87A” composite (head grade of 0.10-0.13% Cu) and resulted in copper concentrates grading as high as 29% Cu at 86% copper recovery (Lakefield test No. 250C). The gold grade of the flotation concentrate was 234g/t Au, and gold recovery was 71%. Although gold recovery was not optimised, it is suggested that a combination of the high pH and sodium sulphite flotation flowsheet and cyanidation of the flotation tails could yield both saleable copper concentrate and high gold recoveries. Further testwork would be required to demonstrate this for lower grade Troilus composites.

Pilot Plant Gravity & Flotation Testwork

The Lakefield 1993 pilot plant testwork was conducted on a higher-grade pilot plant composite. This composite graded 0.19% Cu and 2.06g/t Au compared to 0.16% Cu and 2.11g/t Au for the Zone 87 high grade composite. A total of 18 pilot plant campaigns were executed with primary grind size ranging from 80% passing 40 to 170µm. From PP10 onwards, the primary grind was maintained at 80% passing 85µm. The flowsheet employed for the pilot plant program is shown in Figure 13-3 below.

Figure 13-3: Lakefield 1993 Pilot Plant Flowsheet



The following reagents dosages were reported by Lakefield:

- potassium amyl xanthate (PAX) collector 122 g/t
- aero3477 gold promoter 71 g/t
- lime 203 g/t
- MIBC frother 29 g/t

For the 1993 feasibility study, data from pilot plant campaigns PP16 and PP17 was used for process design due to the stable operating conditions, circuit configuration and relatively consistent head grades. The average metallurgical performance for pilot plant runs PP16 and PP17 is summarised in Table 13-7 below:

Table 13-7: PP16 and PP17 Average Performance

Item	Value
Gold Headgrade, g/t	1.95 g/t
Copper Headgrade, %	0.16%
Primary Circuit Gravity Recovery, % Au	37.5
Regrind Circuit Gravity Recovery, % Au	29.4
Flotation Concentrate Recovery, %Au	21.5
Total Recovery, % Au	88.4
Flotation Concentrate Copper Grade, % Cu	15.3
Flotation Concentrate Copper Recovery, %	89.6

On average, total copper and gold recoveries were 90% and 88% respectively. 67% of the gold recovery was attributed to the combined gravity circuits (Knelson in the ball mill circulating load and Falcon on the cleaner 1 tails) and a further 22% of the gold recovery reported to the copper-gold flotation concentrate. This concentrate graded 15% Cu and 45g/t Au, which is somewhat low grade in copper concentrate grade terms. Higher concentrate grades were produced in earlier pilot plant runs (>20% Cu) but this came at the expense of gold recovery (Table 13-8).

Table 13-8: 1993 Lakefield Pilot Plant PP4, 5, 7 and 11 Metallurgical Performance

Pilot Plant	Grind P_{80}., µm	Cu Head Grade, %	Au Head Grade, g/t	Float Conc Grade, % Cu	Float Conc Grade, g/t Au	Float Conc Cu Rec., %	Float Conc Au Rec., %	Gravity Au Rec., %*	Total Au Rec., %
PP4, 5, 7, 11 Average	77	0.21	2.06	21.6	41.0	**90.5**	18.4	66.2	**84.6**

13.2.3 Metallurgical Testwork, Lakefield, 2003 (J4 Zone)

Inmet Mining contracted Lakefield Research to conduct bench scale gravity and flotation testwork on three composites from the J4 Zone in 2003. At the time, the samples submitted were labelled by Inmet as “future production ore”. Whether or not this material was actually mined and processed in the ensuing years is not known.

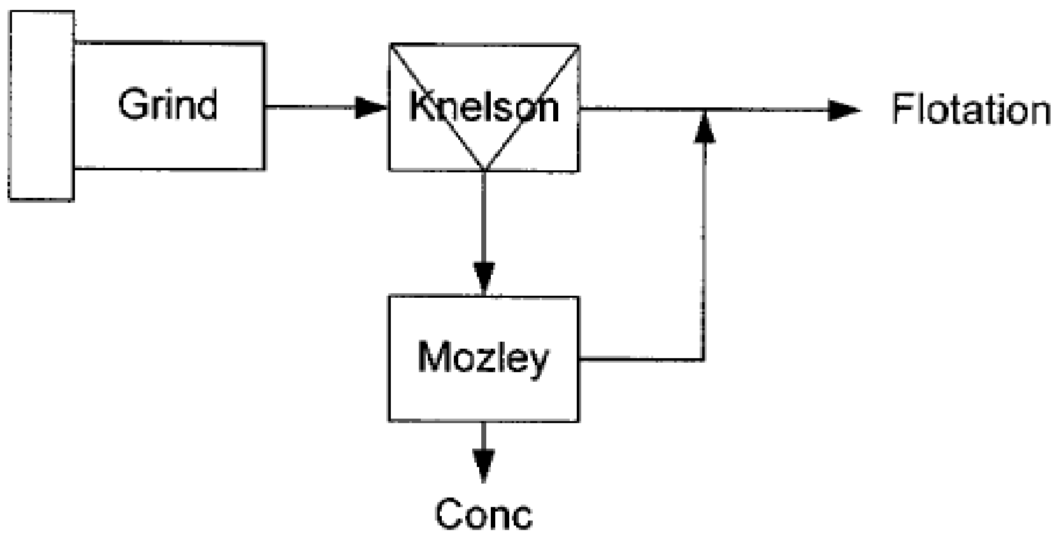
All three composites had similar gold head grades (0.80-1.07g/t Au) and copper head grades were all below 0.1% Cu. Despite the relatively low copper head grades, the composites were identified as low, medium, and high grade based on copper head grade, as summarised in Table 13-9 below:

Table 13-9: J4 Zone Composites (2003) Head Grades

Composite ID	Au g/t	Cu %	S %
J4 Low Grade	1.00	0.023	0.71
J4 Medium Grade	1.07	0.054	1.02
J4 High Grade	0.80	0.096	0.76

Single pass gravity concentration via Knelson concentrator was conducted on a number of batch tests ahead of rougher flotation. The Knelson concentrate was further upgraded via Mozley gravity table as seen in Figure 13-4.

Figure 13-4: Gravity Flowsheet (Lakefield 2003)



The average gold recoveries to Mozley concentrate for these tests are summarised in Table 13-10 below.

Table 13-10: J4 Zone (2003) Summary of Average Gravity Recoverable Gold Performance

Composite	Gravity Conc. Weight %	Gravity Conc. Grade, Au g/t	Gravity Conc. Au Recovery, %	Au Average Head Grade, g/t
J4 Medium	0.017	1660	33	0.79
J4 Low	0.039	863	31	0.79
J4 High	0.020	1453	32	0.93

Gold recovery to gravity concentrate was similar for all composites and ranged from 31-33%. Low mass pull to gravity concentrate was observed (0.017-0.039%), resulting in relatively high-grade gravity concentrates ranging from 900g/t to 1700g/t Au, a range that is more representative of full scale gravity concentrator performance than typical lab work. Grind versus recovery testwork between 80% -94µm and 80% -140µm did indicate that gravity recovery of gold increases with fineness of grind, within the range of 25% to 44%.

Locked cycle flotation testing was conducted on the three composites using the plant conditions from the Troilus mill at the time (i.e.: including gravity concentration). Relatively high dosages of PAX (potassium amyl xanthate) collector were employed, considering the low head grades. This, combined with fine rougher concentrate regrinding, resulted in all tests experiencing instability issues and large circulating loads of pyrite. The conditions for this work included a primary grind of 80% 100 to 115µm, no lime addition to the mill, 25g/t PAX and 25g/t SPRI added to the rougher, pH maintained at 10.7 with lime in the rougher, stainless steel regrinding of rougher concentrate to 15-20µm (with no lime addition), high pH (11.0) in the cleaners using lime, three stages of copper cleaning, with a cleaner scavenger for the first cleaning stage. The flowsheet is shown below in Figure 13-5 and the results are summarized in Table 13-11, Table 13-12, and Table 13-13 below.

Figure 13-5: J4 Zone (2003) Locked Cycle Test Flowsheet

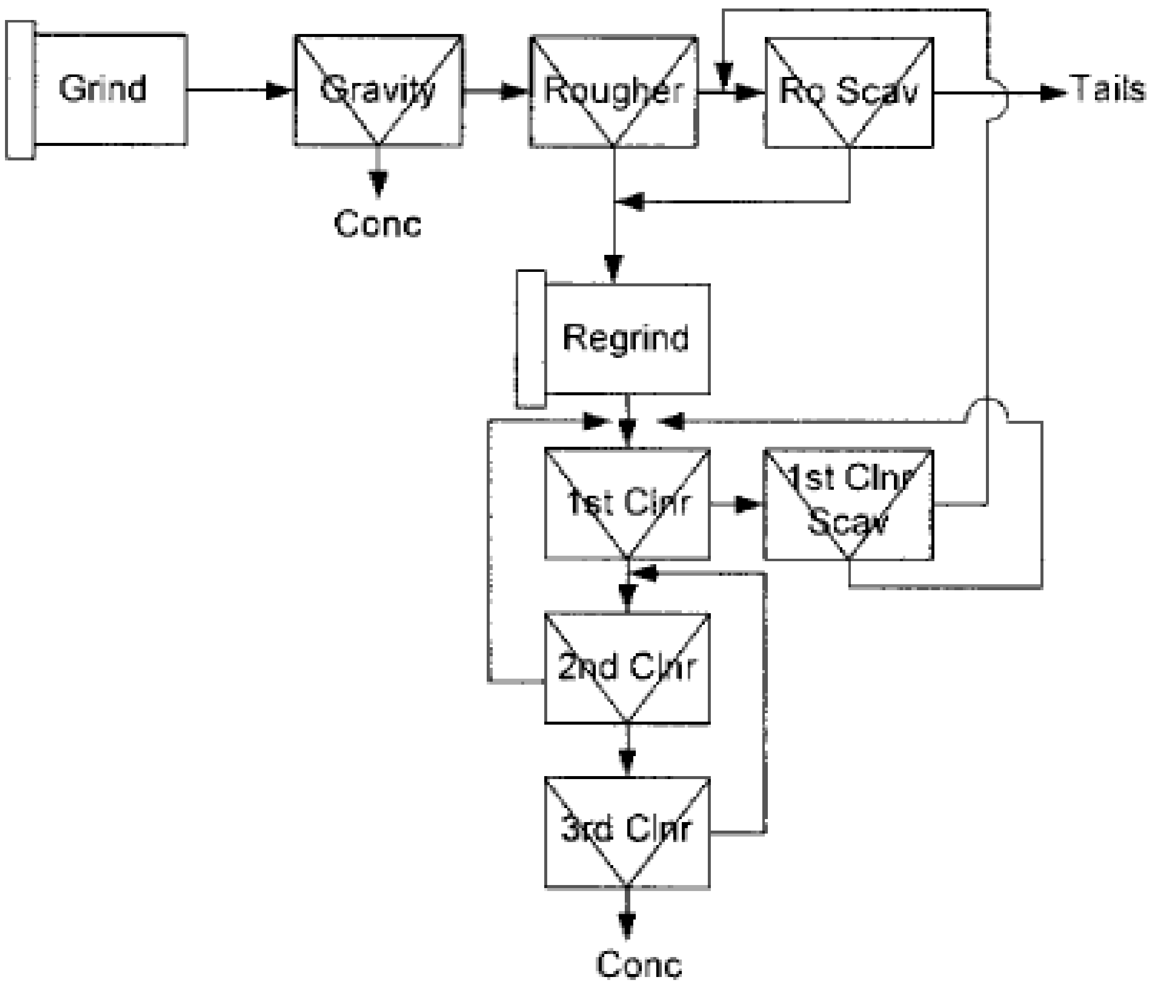


Table 13-11: J4 Zone (2003) Medium Grade LCT Results

Product	Weight %	Cu Grade, %	Au Grade, g/t	Cu Recovery, %	Au Recovery, %
Gravity Conc	0.019		955		26
Cu 3rd Cleaner Conc	0.86	5.81	49.6	93	60
Combined Conc.	0.88	5.81	69.2	**93**	**86**

Table 13-12: J4 Zone (2003) Low Grade LCT Results

Product	Weight %	Cu Grade, %	Au Grade, g/t	Cu Recovery, %	Au Recovery, %
Gravity Conc	0.062		513		36
Cu 3rd Cleaner Conc	0.29	6.47	143.8	83	47
Combined Conc.	0.35	6.47	208.8	**83**	**83**

Table 13-13: J4 Zone (2003) High Grade LCT Results

Product	Weight %	Cu Grade, %	Au Grade, g/t	Cu Recovery, %	Au Recovery, %
Gravity Conc	0.019		1323		30
Cu 3rd Cleaner Conc	0.54	16.0	82.7	93	53
Combined Conc.	0.56	16.0	124.8	**93**	**83**

Although copper and gold recoveries to combined concentrate were acceptable at 83-93% and 83-86% respectively, the concentrate grades were poor for the low and medium grade composites at 6.5% Cu and 5.8% Cu, respectively. At 16% Cu concentrate grade, the final concentrate grade achieved for the high grade composite was in line with previous pilot plant and bench scale testwork conducted at Lakefield in 1993.

In addition to the standard gravity-flotation circuit tested above, two alternative flotation and cyanidation flowsheets were explored on the medium grade composite. These flowsheets involved cyanidation of the cleaner 1 flotation tails as well as rougher flotation followed by regrinding and cyanidation of the reground rougher concentrate. Total gold recoveries were lower ay 79% and 75% respectively, and these flowsheets were not investigated further.

13.2.4 Cyanidation Testwork, COREM, 2019

A total of six composites were subjected to conventional cyanidation bottle roll testing at coarse crush and finer milled particle sizes. Three composites from the J4 Zone and three composites from the J5 zone were tested, with head grades in low, medium, and high gold-grade categories. A summary of the composites and their head grades is provided in Table 13-14 below:

Table 13-14: Summary of COREM 2019 Composite Head Grades

Hole ID	Interval	Category	Au (g/t)	Cu (%)	Ag (g/t)
TLG-ZJ418-065	27m	High Grade	3.01	0.05	1.25
TLG-ZJ418-066	15m	Avg Grade	1.15	0.06	1.32
TLG-ZJ418-065	18m	Low Grade	0.67	0.04	0.98
TLG-ZJ518-023	10m	High Grade	1.55	0.08	1.04
TLG-ZJ518-023	26m	Avg Grade	0.97	0.05	1.00
TLG-ZJ518-034	10m	Low Grade	0.48	0.11	1.68

Gold grades ranged from 0.48g/t to 3.01g/t with silver content consistently low at ~1g/t. The copper head grades were significantly lower than the samples tested in earlier Lakefield work, ranging from 0.04% to 0.11% Cu.

Due to the low copper and silver head grades, the program naturally focused on gold recovery through conventional cyanidation. A total of four bottle roll tests were conducted on each composite with grind/crush size and NaCN concentration being the main variables tested, with the exception of the high and low grade Zone J5 composites where only three tests were carried out. The crush/grind sizes tested were P_{100} = 2mm, P_{80} = 150µm and P_{80} = 75µm. Each test was conducted on a 2.0kg sample at 40% solids with a residence time of 72 hours. The pH was maintained between 10.5-11.0 with the addition of lime, and the NaCN concentration (high = 1000mg/L, low = 200mg/L) was varied for the coarse crush bottle roll tests for four out of the six composites.

The gold recovery data are summarised in Table 13-15 below.

Table 13-15 – Summary of COREM 2019 Bottle Roll Test Results (gold)

Composite	Grind Size, µm	NaCN, mg/L	Assayed Head, g/t	Calc. Head, g/t	Recovery, %
TLG-ZJ418-065	P_{100}=2000	200	3.01	2.34	**41.0**
	P_{100}=2000	1,000		2.59	**56.3**
	P_{80}=150	1,000		2.32	**85.3**
	P_{80}=75	1,000		3.31	**97.0**
TLG-ZJ418-066	P_{100}=2000	200	1.15	1.18	**42.0**
	P_{100}=2000	1,000		1.29	**58.3**
	P_{80}=150	1,000		0.93	**86.9**
	P_{80}=75	1,000		0.85	**93.9**
TLG-ZJ418-065	P_{100}=2000	200	0.67	0.61	**48.9**
	P_{100}=2000	1,000		0.65	**58.1**
	P_{80}=150	1,000		0.59	**89.4**
	P_{80}=75	1,000		0.79	**92.9**
TLG-ZJ518-023	P_{100}=2000	200	1.55	1.64	**59.6**
	P_{80}=150	1,000		1.48	**78.9**
	P_{80}=75	1,000		1.48	**90.0**
TLG-ZJ518-023	P_{100}=2000	200	0.97	0.83	**47.2**
	P_{100}=2000	1,000		0.83	**47.1**
	P_{80}=150	1,000		0.85	**72.5**
	P_{80}=75	1,000		0.93	**77.6**
TLG-ZJ518-034	P_{100}=2000	200	0.48	0.43	**63.3**
	P_{80}=150	1,000		0.42	**85.0**
	P_{80}=75	1,000		0.47	**86.7**

Gold recovery increases significantly with reduced particle size for all six composites, with the highest gold recovery being achieved at the finest grind (P_{80}=75µm). The average recovery at this grind size was 89.7% (min = 77.6%, max = 97.0%), compared to 83.0% at P_{80}=150µm and 52.2% at P_{100}=2000µm. Silver and copper recoveries were significantly lower for all tests conducted averaging 28.9% and 6.2% respectively at the finer grind size.

13.2.5 Metallurgical Testwork, Kappes Cassiday & Associates, 2020

Coarse Bottle Roll Tests

Two composites, Zone J4 and Zone J5, were subjected to coarse bottle roll cyanidation testwork at Kappes Cassiday & Associates (KCA). Ahead of the tests all material was stage crushed via conventional jaw crusher to 80% passing 6.30mm. In addition to the conventional crush tests, a single bottle roll test was conducted on the J4 zone composite after HPGR (high pressure grinding rolls) crushing to 80% passing 5.39mm. The composite head assays are summarised in Table 13-16 below and it should be noted that the gold head grades are aligned with the low-grade composites tested at COREM prior.

Table 13-16: Summary of KCA Composite Head Grades (J4 and J5 Zones)

Composite ID	Au (g/t)	Ag (g/t)	Cu (%)
J4 Zone	0.611	1.32	0.08
J5 Zone	0.521	1.71	0.09

A total of four bottle roll tests were conducted on each crushed composite under a standard set of conditions to simulate heap leaching (240-hour residence time, 1.0g/L NaCN concentration, 5kg sample and 2 minutes/hr rolling time). Results of this work are summarized in Table 13-17 and Table 13-18 below.

Table 13-17: Summary of J4 Zone Conventional Crush Bottle Roll Test Results

KCA Test No.	Composite	Crush Type	Crush Size P_{80}, mm	Calc. Head Au, g/t	Au Recovery, %	NaCN Consumption, kg/t	Lime Addition, kg/t
85484A	J4 Zone	Conventional	6.30	0.681	40	0.10	0.5
85484B	J4 Zone	Conventional	6.30	0.881	37	0.12	0.5
85485A	J4 Zone	Conventional	6.30	0.683	47	0.12	0.5
88003A	J4 Zone	Conventional	6.30	0.672	41	0.13	0.5
Average				**0.729**	**41**	**0.12**	**0.5**
88004A	J4 Zone	HPGR	5.39	0.547	**53**	0.15	0.5

Table 13-18: Summary of J5 Zone Conventional and HPGR Crush Bottle Roll Test Results

KCA Test No.	Composite	Crush Type	Crush Size P_{80}, mm	Calc. Head Au, g/t	Au Recovery, %	NaCN Consumption, kg/t	Lime Addition, kg/t
85485B	J5 Zone	Conventional	6.30	0.696	35	0.15	0.5
85486A	J5 Zone	Conventional	6.30	0.694	35	0.22	0.5
85486B	J5 Zone	Conventional	6.30	0.718	41	0.15	0.5
88003B	J5 Zone	Conventional	6.30	0.671	37	0.13	0.5
Average				**0.695**	**37**	**0.16**	**0.5**

Gold recovery via bottle roll testing of coarsely crushed material was similar for the J4 Zone and J5 Zone composites at 41% and 37% respectively. When J4 Zone material was subjected to HPGR crushing ahead of leaching, the gold recovery increased to 53% (+12%), albeit at a slightly finer crush size. Further investigation would be required to determine whether HPGR crushing ahead of leaching consistently results in higher gold recovery compared to conventional crushing.

A subsequent round of testwork was carried out at KCA in 2020 on four additional composite samples; 87OP, 87UG, 87S and SW. The 87UG, 87S and SW composites in this program were the original material for the comminution testwork conducted at Hazen Research in the same year. In this phase of work at KCA, all composites were coarse-crushed followed by HPGR crushing to produce feed for coarse bottle roll, milled bottle roll and flotation testwork. Direct head assays were not conducted on these composites, therefore the calculated head assays from the various tests were used to calculate an average head grade for each composite. Results are given in Table 13-19 below.

Table 13-19: Summary of 87 and SW Zone Calculated Head Assays

Composite ID	Coarse BRT Calc Head, g/t Au	Milled BRT Calc Head, g/t Au	Leach/Float Calc Head g/t Au	Float/Leach Calc Head g/t Au	Average Calc Head g/t Au	% RSD
87OP	1.987	0.848	1.104	1.275	**1.304**	37
87UG	1.437	0.991	0.940	1.179	**1.137**	20
87S	0.398	0.351	0.542	0.524	**0.454**	21
SW	0.723	0.779	0.756	0.864	**0.781**	8

It should be noted that the relative standard deviations (RSD) for the calculated head data was above 10% for three of the four composites - suggesting that either a procedural error occurred in the sample preparation process, or significant nugget effect is present. This was particularly evident for the 87OP and 87UG composite coarse bottle roll tests.

Coarse bottle roll tests were conducted according to the same procedure applied to the J4 J5 Zone composites discussed above. A summary of these test results is provided in Table 13-20 below.

Table 13-20: Summary of 87 and SW Zone Conventional and HPGR Crush Bottle Roll Test Results

KCA Test No.	Composite ID	Crush Type	Crush Size P_{80}, mm	Calc. Head Au, g/t	Au Recovery, %	NaCN Consumption, kg/t	Lime Addition, kg/t
88032A	87OP	HPGR	6.82	1.987[1]	66	0.45	0.75
88033A	87UG	HPGR	6.52	1.437[2]	59	0.47	0.75
88034A	87S	HPGR	6.77	0.398	47	0.43	0.75
88035A	SW	HPGR	7.26	0.723	48	0.25	0.75
Average				**1.136**	**55**	**0.40**	**0.75**

The average recovery was 55% for these composites which is higher than the J4 and J5 Zone composite recoveries after conventional coarse crushing (41% and 37% respectively), and inline with the 53% recovery achieved for the J4 Zone composite coarse bottle roll test result after HPGR crushing.

Milled Bottle Roll Tests

For this work, each composite was subjected to two milled bottle roll tests, the first at a grind size of 80% -150µm, and the second at a grind of 80% -75µm. Table 13-21 summarizes these results.

[1] Gold recovery in this test should be treated with caution due to the high calculated head grade. No direct head assays were reported by KCA for this composite.
[2] See above comment.

Table 13-21: Summary of 87 and SW Zone Milled Bottle Roll Test Results

KCA Test No.	Composite ID	Grind P_{80}, µm	Leach Res. Time, hrs	NaCN Conc. g/L	Calc. Head Grade, g/t Au	Au Recovery, %	NaCN Consumption, kg/t	Lime Addition, kg/t
88044A	87OP	150	96	1.0	0.848	82	0.46	0.5
88045A	87OP	75	96	1.0	1.104	93	0.64	0.5
88044B	87UG	150	96	1.0	0.991	85	0.57	0.5
88045B	87UG	75	96	1.0	0.940	94	0.80	0.5
88044C	87S	150	96	1.0	0.351	82	0.46	0.5
88045C	87S	75	96	1.0	0.542	89	1.02	0.5
88044D	SW	150	96	1.0	0.779	86	0.50	0.5
88045D	SW	75	96	1.0	0.756	92	0.57	0.5

Gold recovery was consistently (7-11%) higher at the finer grind size for all four composites and significantly higher than the coarse crush (heap leach amenability) bottle roll test results. This further supports the conclusions of earlier testwork: finer grinding results in higher leach recovery.

Froth Flotation

Each of the four composites was subjected to a single rougher flotation test aimed at producing a bulk copper-gold rougher flotation concentrate. Subsequent regrind and cleaner testwork was not conducted, therefore final copper-gold concentrate metallurgical projections can only be made using approximations of cleaner circuit performance. The basic test conditions were as follows:

- primary grind 80% -75µm, no reagents in the mill
- 22% solids flotation pulp density (low density flotation)
- stage addition of lime (60-100g/t) to maintain rougher flotation pH at ~8.0
- 100g/t of potassium amyl xanthate (PAX C3505) collector
- MIBC frother (F500) added as required to maintain a stable froth
- 20 minutes total laboratory flotation time split across 5 rougher stages

A summary of the results for each composite is provided in Table 13-22 below:

Table 13-22: Summary of 87 and SW Zone Run of Mine Rougher Flotation Results

Test ID	Composite ID	Au Head g/t	Cu Head %	Rougher % Mass Pull	Au Grade g/t	Cu Grade %	Au % Recovery	Cu % Recovery
88040	87OP	1.26	0.11	5.1	21.0	2.0	86	95
88041	87UG	1.17	0.17	7.3	14.9	2.3	93	97
88042	87S	0.50	0.10	6.1	7.4	1.5	89	95
88043	SW	0.85	0.05	4.3	15.4	1.0	77	89

Gold and copper recoveries to rougher concentrate averaged 86% and 94% respectively at an average mass pull to rougher concentrate of 5.7% (range of 4.3% to 7.3%). There is some range in gold and copper recovery, but a clear head grade versus metal recovery is not apparent in this small dataset. There does appear to be a relationship between head grade and concentrate grade, with higher head great unsurprisingly leading to higher rougher concentrate grades.

Leaching of Flotation Tails

The final (rougher) flotation tails from the above reported tests were subjected to the standard bottle roll test conditions to determine whether additional gold recovery could be realized. The conditions of these tests broadly followed the previous bottle roll cyanidation tests described earlier:

- no additional grinding of flotation tails i.e. a grind of 80% -75µm was employed
- 1.0g/L NaCN concentration
- 96-hour residence time
- pH maintained between 10.5 to 11.0 with lime

The results for these tests are presented in Table 13-23 below, in conjunction with the rougher flotation results in order to provide a global (flotation plus cyanidation) gold recovery. Copper is not included in this analysis as no further recovery is provided by the cyanidation process.

Table 13-23: Summary of 87 and SW Zone Leaching of Flotation Tails Results

Test ID	Composite ID	Au Flotation Head, g/t	Au Flotation Tails, g/t	Au Float Recovery %	Au Float Tails Leach Recovery %	Combined Au Recovery %
88046A	87OP	1.26	0.21	86	74	**96**
88046B	87UG	1.17	0.10	93	69	**98**
88046C	87S	0.50	0.08	89	65	**96**
88046D	SW	0.85	0.22	77	76	**95**

Gold extractions ranging from 65% (composite 87S) to 76% (composite SW) were obtained, bringing combined rougher flotation, and leaching of rougher flotation tails gold recovery to 95-98% (average 96%).

Flotation of Leach Residues

Although not commonly practised, cyanide leaching of milled run of mine material, followed by flotation of copper sulphides from the leach tails has been successfully trialled on various projects in the past and is technically feasible if residual cyanide is destroyed by one of the normal CN detox processes. In this testwork, the leach residue slurries from the 75µm milled bottle roll tests were filtered, re-pulped with tap water and transferred directly to a Denver D12 flotation cell for rougher flotation testing. Cyanide destruction was not conducted prior to these flotation tests, but it is expected that the majority of the residual free cyanide would be removed during the filtration and washing process. Subsequent regrind and cleaner testwork was not conducted, therefore final copper-gold concentrate metallurgical projections cannot be made. The basic test conditions were as follows:

- no additional grinding prior to flotation: leach tails floated at 80% passing 75µm
- 22% solids flotation pulp density (low density flotation)
- rougher pH target >8.0. No additional lime was required to achieve this for the leach tails
- 100g/t of potassium amyl xanthate (PAX C3505) collector
- MIBC frother (F500) added as required to maintain a stable froth

- 20 minutes total laboratory flotation time split across 5 rougher stages

The data presented below in Table 13-24 shows the total recovery of gold and copper as the sum of the recovery via direct cyanidation followed by flotation of the cyanide tailings into a rougher concentrate. Additional processing of the rougher concentrate will be required and will likely result in some losses of both gold and copper. It should also be noted that copper recovered in the cyanidation step would not be in a payable form.

Table 13-24: Summary of 87 and SW Zone Leach and Rougher Flotation Results

Comp ID	Leach Au Head g/t	Leach Cu Head %	Leach Rec Cu %	Leach Rec Au %	Rougher Mass Pull %	Conc Au Grade g/t	Conc Cu Grade %	Conc Au Rec %	Conc Cu Rec %	Total Au Rec %	Total Cu Rec %
87OP	1.104	0.11	7	93	5.4	0.418	1.8	31	95	**95**	**96**
87UG	0.940	0.16	7	94	7.9	0.293	1.8	41	97	**96**	**97**
87S	0.542	0.09	6	89	6.2	0.283	1.3	30	95	**92**	**95**
SW	0.746	0.04	36	92	5.0	0.276	0.5	23	88	**94**	**93**

Solid-Liquid Separation & Rheology Testwork

A sample of cyanide leach tailings and filtrate was generated from an equally weighted composite of 87OP, 87UG, 87S and SW material, and was sent to Pocock Industrial Inc. in Salt Lake City, USA for flocculent screening, static settling, and rheology testwork. The as received samples were characterised as shown in Table 13-25 below.

Table 13-25: Summary of Leach Tails Characterisation prior to SLS Testwork

Composite ID	pH	Average Liquor SG	Average Solids SG	P_{80} µm	% passing 25µm
88054A	8.6	1.0	2.98	58	33

Flocculant screening tests were completed on small samples of pulp to compare effectiveness in floccule formation, fines capture, liquor release, settling characteristics and approximate dosage levels. The flocculant product selected from screening tests for best performance was SNF AN905 SH, a medium to high molecular weight 5% charge density anionic polyacrylamide. A minimum effective dosage rate was specified at 10-15g/t at a flocculent concentration of 0.1g/L

Static thickening tests were conducted on the samples at various flocculant doses and feed solids concentrations to develop and optimize parameters for conventional type thickener design (Table 13-26). The samples tested showed reasonable settling characteristics, with a flocculant dosage of 15–20g/t (using SNF AN905 SH). The 88054A Material thickened well in the feed solids range of 20% - 25%. Recommended underflow density for the material is in the range of 60-64%, for a standard type conventional thickener, with a minimum recommended unit area of 0.196m^2/mtpd.

Table 13-26: Summary of Recommended Thickener Design Parameters for Leach Tails

Floc Type	Floc Dose g/t	Floc Conc g/L	Max. Thickener Feed Solids %	Min. Unit Area (m^2/mtpd)	Estimated U/F Density %
SNF AN905 SH	15-20 Conv. type	0.1-0.2	20-25 Conv. type	0.196 Conv. type	60-64%

Yield stress versus thickener underflow solids density curves derived from rheology testwork at Pocock indicate an exponential increase in viscosity at solids densities above 60-64%. Therefore it is not recommended to design for thickener underflow densities of above 64%.

13.3 Suitability of Metallurgical Samples

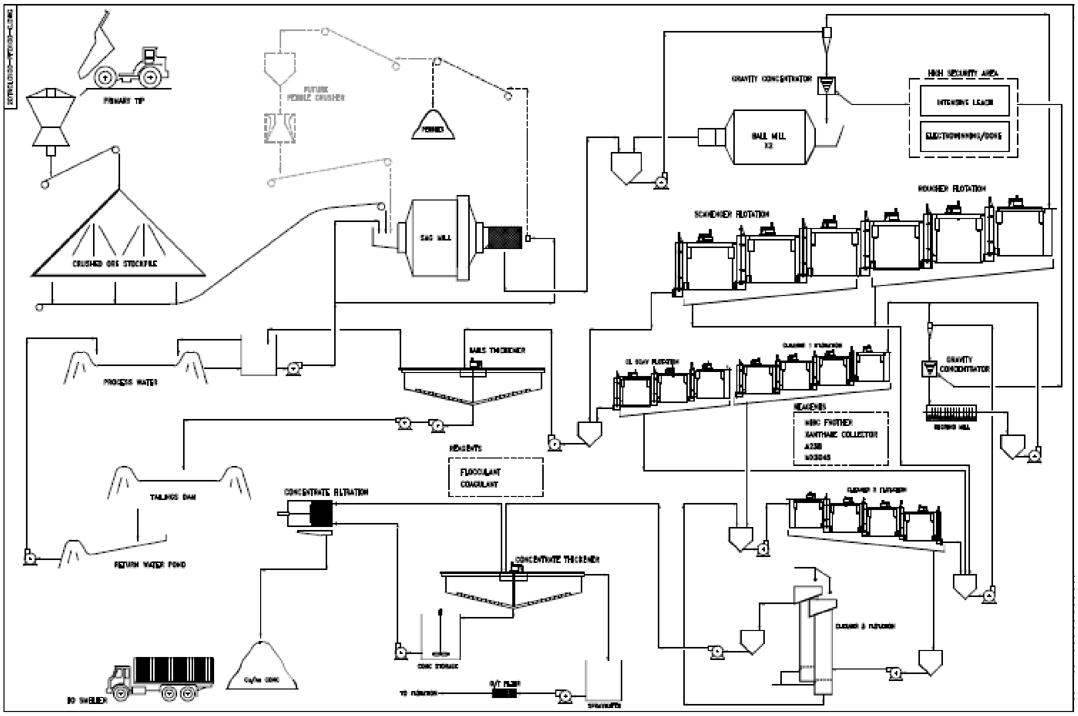
The fairly large body of testwork and operating data that forms part of the metallurgical reference for this project is considered to be suitably diverse (spatially and grade-wise) for a preliminary assessment of project economics. The project consists of several discrete zones, and these have all been characterised to some extent by the testwork. Further testwork should improve the representation of the deposit by considering the most up to date mine plans when selections are made.

Good quality full flowsheet (and even pilot plant) testwork results are available dating back to 1993 and although this work was completed to a high standard by a reputable laboratory, the samples tested at that time were likely taken from zones within the mine plans of that period (1996 to 2010). As these zones are now likely to be fully mined-out, then theoretically the composition of these early samples could differ in grade, mineralogy and metallurgical performance from the zones/volumes selected in the current preliminary mine plans.

13.4 Metallurgical Projection

Various processing options have been considered for the Troilus project as part of this preliminary economic assessment, and a simple gravity plus copper-gold froth flotation approach has been selected as the most beneficial at this time. The proposed flowsheet is described in Section 17 and summarized in Figure 13-6 below.

Figure 13-6: Process Flowsheet



Options to leach the residual gold in flotation tailing slurries, or alternatively to use froth flotation to recover copper from cyanidation residues should be considered in future studies as these may also have economic merit.

A large volume of metallurgical test data has been reviewed for this study, dating back to the early 1990's. In preparing the metallurgical performance predictions required for this economic evaluation, the following factors have been considered and form part of the flowsheet selection methodology:

- Gravity gold recovery from the 1993 and 2003 bench scale testwork on 87 Zone and J4 Zone material (excluding the high-grade pilot plant sample) averaged 40%. Concentrate grades were reported to be >800g/t Au, which is thought to be well within the range for industrial operation and quite suitable feed for intensive cyanidation and doré production. Modern methods of modelling gravity recoverable gold (GRG) content have evolved since the time of historical testwork and it is thought likely that modern E-GRG testwork at coarser grinds, representative of actual plant conditions (gravity concentration would likely be conducted on the cyclone underflow), might result in a slight de-rating of this performance point. For the purposes of this preliminary economic study, AGP has assumed that like historical Troilus operations, gravity concentration would be carried out on the primary ball mill cyclone underflow and also on the regrind mill cyclone underflow streams. A somewhat conservative gravity recovery of 30% for gold and 0% for copper has been used for the economic analysis.

- Recent flotation testwork at Kappes Cassiday & Associates only involved the rougher portion of the circuit (ie. no cleaner circuit data). Gravity concentration, rougher concentrate regrinding and cleaner flotation stages were not included and therefore projections of final concentrate grade and recovery have been estimated using average the gravity assumptions mentioned above, plus cleaner circuit performance predictions from other tests (equivalent to 95% unit recovery in each of 3 stages of cleaning).
- Historical data records from Inmet operations (1997 – 1999) have been referenced, albeit acknowledging that for certain periods of operation the process plant was under production-related pressures that resulted in off-spec (coarse) grinds from the mill and correspondingly low recoveries. For this preliminary economic assessment, AGP has determined that a finer grind should be utilized for the process flowsheet, thereby helping to improve the recovery of copper and gold.

Flotation concentrate grades and recoveries have been derived for the various zones as shown in Table 13-27 below.

Table 13-27: Metallurgical Predictions for Flotation

Zone	Head Grade, % Cu	Conc Grade, % Cu	Recovery, %	Gold Recovery, %
87 Zone	0.09 average	23	90	60
J Zone	0.06 average	12	90	60
SW Zone	when >= 0.13	19	92	if Head Grade >1.2 g/t – 60 if Head Grade <1.2 g/t – 58
SW Zone	when < 0.13	17	90	if Head Grade >1.2 g/t – 60 if Head Grade <1.2 g/t – 58

Copper performance is related to head grade, and in cases where head grade is lower (<0.09% Cu) then reasonable copper concentrate grades become difficult to achieve without sacrificing copper and gold recovery. In these instances (J Zone for example) concentrate grade is sacrificed in return for recovery. This can be tolerated commercially as a result of the supplemental gold grades in these low-grade copper products, and the fact that for several years it can be blended with higher grade products (87 UG zone or SW Zone).

The metallurgical performance of silver is not reported comprehensively in the metallurgical testwork, and therefore a somewhat conservative assumption of 40% recovery has been allowed across the board for this metal.

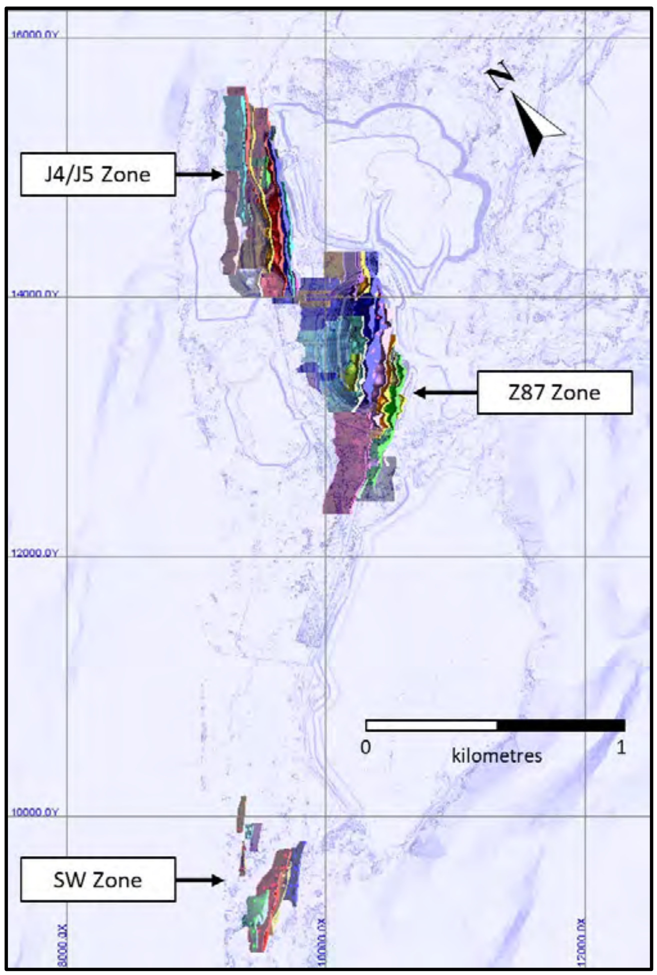
14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

This section discloses the mineral resources 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., Senior Resource Geologist for AGP. The effective date of this mineral resource is 20 July 2020.

The current mineral resources for Troilus includes open pit resources for Z87, J4/J5, and SW Zones and underground resources for Z87 Zone. The resource estimate has been prepared using interpreted mineralized domains at the three deposits that comprise the Project (Figure 14-1).

Figure 14-1: Z87, J4/J5 and SW Zones of the Troilus Project



Source: AGP (2020)

14.1.1 AGP Validation, Z87 Zone, J4/J5 Zone

AGP validated the resource estimates for the Z87 Zone and J4/J5 Zone originally estimated by Roscoe Postle Associates Inc. (RPA). The resource estimates were described in a report authored by Luke Evans dated December 20, 2020, titled "Technical Report on the Troilus Gold-Copper Project Mineral Resource Estimate, Quebec, Canada" (RPA (2019b). Since the report was published, there was no further work conducted on the Z87 Zone and J4/J5 Zone.

The review focused on the drill hole database validation, assay validation against the laboratory certificate, wireframe validation, and grade interpolation. The purpose of this review is to validate the current model and provide recommendations for improvement of the grade estimate and/or improve the model classification. AGP accepts the validity of the model, which was not re-interpolated; therefore, the text and tables were extracted from the previous NI 43-101 report from RPA (2019b).

All mineral resources described herein have been reported within updated constraining shells.

14.2 Mineral Resources Summary

The resource estimates were completed using Geovia GEMS™ 6.8.3 resource estimation software. The coordinate system used a mine grid, rotated approximately 35° Az from the UTM coordinate NAD83 system. The Z87 and J4/J5 resource estimate used a block matrix of 5m x 5m x 5m and the SW Zone used a block matrix of 10m x 10m x 10m. The blocks model grades were estimated using ordinary kriging interpolation method using 2m capped composites. Metal grades were capped post compositing for Z87, and J4/J5 Zones, and prior to compositing for the SW Zone. Capping levels vary based on mineralized domain, however, and not all domains required capping or metal grades.

The mineral resources amenable to open pit extraction are reported within optimized constraining shells for each mineralized zone at a 0.3 gpt AuEQ cut off grade; and mineral resources amenable to underground at the Z87 Zone are reported based on a 0.9 gpt AuEQ cut-off grade for contiguous blocks, below 4900 m elevation; and below the constraining shell for J4/J5 Zone.

The optimized constraining shells were developed for each deposit by AGP using Hexagon Mining MineSight 3D software and incorporates metal recovery, geotechnical parameters, and assumed 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 Mineral Resources for the combined mineral resources amenable to open pit and underground resources for the Troilus Project.

Table 14-1: Mineral Resources for the Troilus Project; combined open pit and underground resources

Classification	Tonnes (,000t)	Grade				Contained Metal			
		Au (gpt Au)	Cu (% Cu)	Ag (gpt Ag)	AuEQ (gpt AuEQ)	Au (Moz)	Cu (Mlbs)	Ag (Moz)	AuEQ (M oz)
Indicated	177.3	0.75	0.08	1.17	0.87	4.30	322.60	6.66	4.96
Inferred	116.7	0.73	0.07	1.04	0.84	2.76	189.73	3.91	3.15

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 gpt AuEQ where the metal equivalents were calculated as follows:
  - Z87 Zone AuEq = Au grade + 1.2566 * Cu grade + 0.0103 * Ag grade
  - J4/J5 Zone AuEq = Au grade + 1.2979 * Cu grade + 0.0108 * Ag grade
  - SW Zone AuEq = Au grade + 1.2768 * Cu grade + 0.0106 * Ag grade
- Metal prices for the AuEQ formulas are: $US 1,600/ oz Au; $3.25/lb Cu, and $20.00/ oz Ag; with an exchange rate of US$1.00: CAD$1.30
- Metal recoveries for the AuEQ formulas are:
  - Z87 Zone 83% for Au recovery, 92% for Cu recovery and 76% for Ag recovery
  - J4J5 Zone 82% for Au recovery, 88% for Cu recovery and 76% for Ag recovery
  - Z87 Zone 82.5% for Au recovery, 90% for Cu recovery and 76% for Ag recovery
- Underground cut-off grade is 0.9 AuEQ at Z87 Zone and J4/J5 Zone, contiguous blocks below constraining shell
- Capping of grades varied between 2.00 g/t Au and 26.00 g/t Au; between 1.00 g/t Ag and 20.00 g/t Ag on raw assays; and 1.00 %Cu on raw assays
- The density varies between 2.72 g/cm^3 and 2.91 g/cm^3 depending on mineralized zone

14.2.1 Open Pit Mineral Resources

The mineral resources for the Troilus Project deposit amenable to open pit extraction at a 0.3 gpt AuEQ cut-off grade are: an Indicated Resource of 164.2 Mt at 0.68 g/t Au, 0.08 %Cu, 1.20 gpt Ag and 0.80 gpt AuEQ; and an Inferred Resource of 101.2 Mt at 0.60 g/t Au, 0.07 %Cu, 1.12 gpt Ag and 0.70 gpt AuEQ.

Table 14-2 presents the Mineral Resources amenable to open pit extraction.

Table 14-2: Open Pit Mineral Resources for Troilus Project at a 0.3 gpt AuEQ Cut-off Grade – All Deposits

Classification	Tonnes (,000t)	Grade				Contained Metal			
		Au (gpt Au)	Cu (% Cu)	Ag (gpt Ag)	AuEQ (gpt AuEQ)	Au (Moz)	Cu (Mlbs)	Ag (Moz)	AuEQ (M oz)
Z87 Zone									
Indicated	84.6	0.79	0.09	1.39	0.92	2.15	169.54	3.77	2.50
Inferred	32.7	0.60	0.07	1.50	0.70	0.63	49.34	1.57	0.73
J4/J5 Zone									
Indicated	79.6	0.57	0.07	1.00	0.67	1.47	115.16	2.55	1.71
Inferred	45.9	0.55	0.07	0.96	0.65	0.82	65.94	1.42	0.96
SW Zone									
Inferred	22.6	0.70	0.07	0.89	0.80	0.51	35.73	0.65	0.58
TOTALS									
Indicated	164.2	0.68	0.08	1.20	0.80	3.62	284.69	6.32	4.21
Inferred	101.2	0.60	0.07	1.12	0.70	1.95	151.01	3.65	2.27

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 gpt AuEQ
- AuEQ equivalents were calculated as follows:
  - Z87 Zone AuEq = Au grade + 1.2566 * Cu grade + 0.0103 * Ag grade
  - J4/J5 Zone AuEq = Au grade + 1.2979 * Cu grade + 0.0108 * Ag grade
  - SW Zone AuEq = Au grade + 1.2768 * Cu grade + 0.0106 * Ag grade
- Metal prices for the AuEQ formulas are: $US 1,600/ oz Au; $3.25/lb Cu, and $20.00/ oz Ag; with an exchange rate of US$1.00: CAD$1.25
- Metal recoveries for the AuEQ formulas are:
  - Z87 Zone 83% for Au recovery, 92% for Cu recovery and 76% for Ag recovery
  - J4J5 Zone 82% for Au recovery, 88% for Cu recovery and 76% for Ag recovery
  - Z87 Zone 82.5% for Au recovery, 90% for Cu recovery and 76% for Ag recovery
- Capping of grades varied between 2.00 g/t Au and 26.00 g/t Au; between 1.00 g/t Ag and 20.00 g/t Ag on raw assays; and 1.00 %Cu on raw assays
- The density varies between 2.72 g/cm^3 and 2.91 g/cm^3 depending on mineralized zone or domain

14.2.2 Underground Mineral Resources

The mineral resources for the Troilus Project deposit amenable to underground extraction are: An Indicated Resource of 13.1 Mt at 1.61 g/t Au, 0.13 %Cu, 0.81 gpt Ag and 1.79 gpt AuEQ; and an Inferred Resource of 15.5 Mt at 1.62 g/t Au, 0.1 %Cu, 0.52 gpt Ag and 1.77 gpt 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 gpt AuEQ Cut-off Grade – Z87 Zone

Classification	Tonnes (,000t)	Grade				Contained Metal			
		Au (gpt Au)	Cu (% Cu)	Ag (gpt Ag)	AuEQ (gpt AuEQ)	Au (Moz)	Cu (Mlbs)	Ag (Moz)	AuEQ (M oz)
Z87 Zone									
Indicated	13.1	1.61	0.13	0.81	1.79	0.68	37.90	0.34	0.75
Inferred	13.5	1.70	0.12	0.37	1.85	0.74	34.48	0.16	0.80
J4/J5 Zone									
Indicated	0.01	1.03	0.03	0.47	1.07	0.0002	0.01	0.0001	0.0003
Inferred	2.0	1.06	0.10	1.55	1.21	0.07	4.24	0.10	0.08
TOTALS									
Indicated	13.1	1.61	0.13	0.81	1.79	0.68	37.91	0.34	0.75
Inferred	15.5	1.62	0.11	0.52	1.77	0.81	38.72	0.26	0.88

Notes:

- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
- Summation errors may occur due to rounding
- Underground cut-off grade is 0.9 gpt AuEQ
- AuEQ equivalents were calculated as follows:
  - Z87 Zone AuEq = Au grade + 1.2566 * Cu grade + 0.0103 * Ag grade
  - J4/J5 Zone AuEq = Au grade + 1.2979 * Cu grade + 0.01083 * Ag grade
- Metal prices for the AuEQ formulas are: $US 1,600/ oz Au; $3.25/lb Cu, and $20.00/ oz Ag; with an exchange rate of US$1.00: CAD$1.25.
- Metal recoveries for the AuEQ formulas are:
  - Z87 Zone 83% for Au recovery, 92% for Cu recovery and 76% for Ag recovery
  - J4/J5 Zone 82% for Au recovery, 88% for Cu recovery and 76% for Ag recovery
- Capping of grades varied between 5.00 g/t Au and 26.00 g/t Au: between 10.00 g/t Ag and 20.00 g/t Ag on raw assays
- The density of the mineralized domains at Z87 Zone is 2.86 g/cm^3: and 2.77 and 2.78 g/cm^3 at J4/J5 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 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.

14.3 Database

The Troilus drill hole database for Z87 and J4/J5 Zones contains 805 surface diamond drill holes with a total length of 199,226 m. Of these drill holes, 724 drillholes, totalling 184,762 m, are attributed to the Z87 and J4/J5 Zones. The database for the SW Zone contains 87 surface diamond drill holes with a total length of 16,121 m.

Table 14-4 presents a summary of drill holes in the database and drill holes used in the estimation of resources for the three deposits.

Table 14-4: Summary of Drill Hole Database for the Project; up to February 2020

Deposit	Year	All Drill Holes		Drill Holes used for Resources	
		Number	Metres	Number	Metres
Z87 Zone	< 2018	387	86,420.05	374	84,189.58
	2018	38	23,226.58	38	23,226.58
	2019	22	8,863.00	21	8,767.00
	Total	**447**	**118,509.63**	**433**	**116,183.16**
J4/J5 Zone	< 2018	184	28,801.30	173	27,538.71
	2018	47	13,290.40	47	13,290.40
	2019	46	24,161.00	46	24,161.00
	Total	**277**	**66,252.70**	**266**	**64,990.11**
SW Zone	< 2018	63	7,401.64	12	2,203.98
	2019	7	2,682.00	7	2,682.00
	2020	17	6,037.50	16	5,521.50
	Total	**87**	**16,121.14**	**35**	**10,407.48**

AGP received the database as a GEMS project for all deposits of the Project, as well as the Geotic database export files for the SW Zone. The database was made up of several tables that included, but are not limited to, collar, survey, assay, composite, and density. The GEMS validation tool was used to verify the databases for the collar, survey, and assay tables; there were no errors found.

All data received was in the local grid coordinate system for the Troilus Project, which is rotated approximately 55 degrees to the east of magnetic north. Coordinates for all drill holes are available in NAD83 datum in the Geotic database. As described in Section 12 of this report, AGP reviewed approximately 15% of the assay database distributed over the three deposits comparing the results from the assay certificates issued by the laboratory. No errors found. The author is of the opinion that the database is adequate for the purposes of mineral resource estimation for the Project.

14.4 Z87 Zones

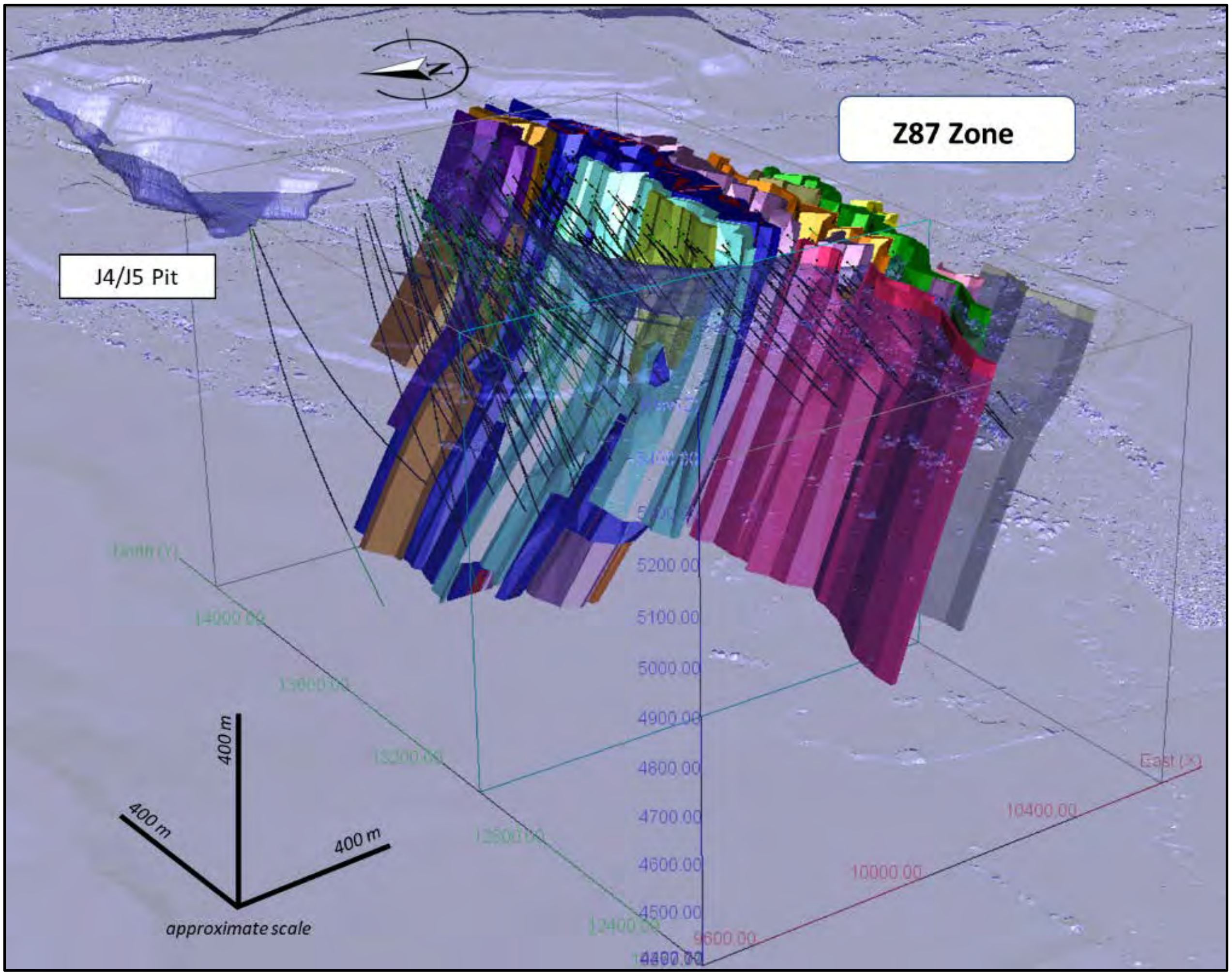
14.4.1 Geological Models

The mineralized zones at Z87 Zone were interpreted by Troilus personnel. The interpreted wireframes were completed using conventional polylines on vertical sections defined along 25 m spaced sections. The polylines capture a minimum nominal grade of 0.3 gpt AuEQ to a minimum width of 4 m. One higher grade domain was created to capture a minimum nominal grade of 0.9 gpt AuEQ. The polylines were 'wobbled' (GEMS function) and snapped to drill hole intercepts. The polylines were then joined together using tie lines in order to create 3D solid wireframes. The mineralized envelops were created above topography and extended approximately 600 m to 900 m below surface. A total of 16 wireframes were created for the Z87 Zone. The high grade domain was subdivided into three subdomains.

AGP reviewed the Z87 Zone wireframes and found no errors. AGP agrees that these wireframes are suitable to estimate resources for the Z87 Zone. AGP notes that the wireframes exhibit some zig-zag, or sawtooth, shapes when viewed in plan view. While these shapes are not overly exaggerated, it is recommended that these wireframes be revisited to adjust along plan views in plan view. AGP anticipates these changes would not have affect the current mineral resources but may show a better representation of the mineralization.

Figure 14-2 shows the mineralized wireframes for the Z87 Zone.

Figure 14-2: Mineralized Domains – Z87 Zone



Source : AGP (2020)

Table 14-5 lists the mineralized domain wireframes and subdomains for the Z87 Zone.

Table 14-5: Domains and Subdomains – Z87 Zone

Mineralized Zone	Rock Type	Subdomain Zone	Rock Type	Comment
10	10	1001	1001	0.9 gpt AuEQ subdomain
		1002	1002	0.9 gpt AuEQ subdomain
		1003	1003	0.9 gpt AuEQ subdomain
11	11			
12	12			
13	13			
14	14			
15	15			
16	16			
17	17			
18	18			
19	19			
20	20			
21	21			
22	22			
23	23			
24	24			
25	25			

14.4.2 Exploratory Data Analysis

Raw Assays

The drill hole database for Z87 Zone data, consists of 387 drill holes and 66,195 assay values for each metal: gold, copper, and silver. Any assay values reported below detection limit were assigned half the detection limit for statistical analysis and grade estimation. Any missing values were assigned a zero. Table 14-6 presents the overall statistics for the Z87 Zone. Out of this total, 15,270 assay values are captured by the mineralized domains.

Table 14-6: Descriptive Statistics of Raw Assays within Mineralized Domains – Z87 Zone

	Au (gpt)	Cu (%)	Ag (gpt)	Length (m)
Count	25525	25525	25525	25525
Minimum	0.001	0.00	0.00	0.03
Maximum	133.70	11.27	259.90	20.00
Mean	0.46	0.09	1.19	1.25
Median	0.12	0.04	0.60	1.00
Std. Deviation	1.70	0.19	3.79	0.54
CV	3.73	2.06	3.18	0.43

Table 14-7 to Table 14-9 presents the descriptive statistics for raw gold, copper, and silver assays, respectively, by mineralized domain.

Table 14-7: Descriptive Statistics for Raw Gold Assays (gpt Au) – Z87 Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
1001	5,987	0.00	103.01	1.65	1.02	3.09	1.88
1002	430	0.00	32.35	1.65	0.94	2.72	1.65
1003	78	0.04	10.69	2.11	1.17	2.50	1.18
11	6,406	0.00	36.70	0.56	0.35	1.00	1.80
12	399	0.00	10.70	0.44	0.25	0.84	1.93
16	223	0.00	13.90	0.33	0.19	1.00	3.03
18	298	0.00	29.07	0.81	0.29	2.13	2.64
20	1,621	0.00	19.94	0.62	0.36	0.95	1.52
21	288	0.00	4.07	0.53	0.36	0.56	1.05
13	4,249	0.00	133.70	0.82	0.32	3.69	4.49
14	2,842	0.00	87.40	0.79	0.36	2.45	3.09
15	26	0.03	1.81	0.55	0.35	0.52	0.94
17	1,907	0.00	35.04	0.63	0.30	1.65	2.62
19	194	0.00	17.68	0.70	0.33	1.63	2.32
22	76	0.00	3.09	0.36	0.24	0.41	1.16
23	34	0.00	2.62	0.47	0.29	0.57	1.20
24	34	0.03	1.49	0.40	0.32	0.36	0.90
25	343	0.00	12.20	0.54	0.31	0.91	1.67

Table 14-8: Descriptive Statistics for Raw Copper Assays (%Cu) – Z87 Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
1001	5,987	0.00	9.58	0.16	0.10	0.24	1.50
1002	430	0.00	2.89	0.21	0.14	0.26	1.20
1003	78	0.01	1.05	0.14	0.09	0.17	1.20
11	6,406	0.00	3.33	0.06	0.04	0.10	1.59
12	399	0.00	0.75	0.10	0.06	0.10	1.09
16	223	0.00	1.01	0.09	0.07	0.10	1.10
18	298	0.00	0.29	0.03	0.02	0.04	1.22
20	1,621	0.00	0.83	0.04	0.02	0.05	1.46
21	288	0.00	0.20	0.02	0.01	0.03	1.28
13	4,249	0.00	10.00	0.10	0.04	0.24	2.27
14	2,842	0.00	0.92	0.05	0.03	0.06	1.30
15	26	0.01	0.14	0.05	0.03	0.04	0.77
17	1,907	0.00	11.27	0.08	0.03	0.30	3.88
19	194	0.00	2.00	0.04	0.02	0.15	3.34
22	76	0.01	0.36	0.09	0.06	0.07	0.86
23	34	0.01	0.19	0.05	0.04	0.04	0.83
24	34	0.00	0.14	0.03	0.02	0.03	0.88
25	343	0.00	0.83	0.03	0.02	0.06	1.87

Table 14-9: Descriptive Statistics for Raw Silver Assays (gpt Ag) – Z87 Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
1001	5,987	0.00	109.50	1.51	0.80	3.69	2.44
1002	430	0.00	52.00	3.23	1.40	6.04	1.87
1003	78	0.00	4.13	0.24	0.00	0.64	2.65
11	6,406	0.00	259.90	0.92	0.46	3.70	4.04
12	399	0.00	15.10	2.12	1.40	2.44	1.15
16	223	0.00	11.30	1.33	0.90	1.58	1.19
18	298	0.00	9.50	0.61	0.30	1.00	1.64
20	1,621	0.00	253.56	0.70	0.20	6.42	9.15
21	288	0.00	12.20	0.53	0.20	1.02	1.91
13	4,249	0.00	122.00	1.16	0.40	3.71	3.20
14	2,842	0.00	159.54	1.32	0.70	3.60	2.72
15	26	0.25	7.20	1.28	0.95	1.47	1.15
17	1,907	0.00	59.20	1.10	0.56	2.31	2.09
19	194	0.00	39.20	1.13	0.65	2.92	2.59
22	76	0.25	7.70	1.39	1.10	1.28	0.92
23	34	0.25	2.30	0.78	0.70	0.56	0.72
24	34	0.25	5.70	1.80	1.45	1.53	0.85
25	343	0.00	14.50	0.65	0.35	1.11	1.71

Capping Analysis

Capping analysis was carried out on each mineralized domain for gold, copper, and silver. Capping was applied to gold and silver assay values in several mineralized domains. Not all domains required capping levels. No capping was applied to copper assay values. AGP reviewed the capping levels by domain using histogram and disintegration plots and found the capping levels to be reasonable and adequate.

Table 14-10 presents the capping levels for gold and silver, by domain, for the Z87 Zone. Table 14-11 and Table 14-12 present the descriptive statistics for capped gold and silver assay values, respectively.

Table 14-10: Capping Levels – Z87 Zone

Domain	Au (gpt)	% Loss	Cu (%)	% Loss	Ag (gpt)	% Loss
1001	26.00 (16)	2.8	-		20.00 (29)	7.4
1002						
1003						
11	10.00 (7)	2.0	-		15.00 (16)	7.1
12	5.00 (5)	5.6	-		10.00 (4)	0.9
16	5.00 (2)	12.0	-		-	
18	8.00 (2)	11.0	-		-	
20	7.00 (3)	1.6	-		10.00 (8)	25.0
21	-		-		-	
13	20.00 (14)	13.0	-		20.00 (13)	7.8
14	15.00 (7)	6.7	-		15.00 (10)	5.7
15	-		-		-	
17	10.00 (8)	7.4	-		12.00 (10)	5.1
19	6.00 (2)	13.0	-		10.00 (1)	13
22	-		-		-	
23	-		-		-	
24	-		-		-	
25	5.00 (1)	3.9	-		10.00 (1)	2

(X) – number of assays capped

Table 14-11: Descriptive Statistics for Capped Raw Assay Values for Gold (gpt Au) – Z87 Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
1001	5987	0.00	20.00	1.41	0.80	2.11	1.50
1002	430	0.00	20.00	2.87	1.40	4.03	1.41
1003	78	0.00	4.13	0.24	0.00	0.63	2.65
11	6406	0.00	15.00	0.85	0.46	1.40	1.64
12	399	0.00	10.00	2.10	1.40	2.36	1.12
16	223	0.00	11.30	1.33	0.90	1.58	1.19
18	298	0.00	9.50	0.61	0.30	1.00	1.64
20	1621	0.00	10.00	0.53	0.20	1.01	1.92
21	288	0.00	12.20	0.53	0.20	1.02	1.91
13	4249	0.00	20.00	1.07	0.40	2.11	1.97
14	2842	0.00	15.00	1.25	0.70	1.77	1.42
15	26	0.25	7.20	1.28	0.95	1.47	1.15
17	1907	0.00	12.00	1.05	0.56	1.61	1.54
19	194	0.00	10.00	0.98	0.65	1.18	1.21
22	76	0.25	7.70	1.39	1.10	1.28	0.92
23	34	0.25	2.30	0.78	0.70	0.56	0.72
24	34	0.25	5.70	1.80	1.45	1.53	0.85
25	343	0.00	10.00	0.64	0.35	0.97	1.51

Table 14-12: Descriptive Statistics for Capped Raw Assay Values for Silver (gpt Ag) – Z87 Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
1001	5987	0.00	20.00	1.41	0.80	2.11	1.50
1002	430	0.00	20.00	2.87	1.40	4.03	1.41
1003	78	0.00	4.13	0.24	0.00	0.63	2.65
11	6406	0.00	15.00	0.85	0.46	1.40	1.64
12	399	0.00	10.00	2.10	1.40	2.36	1.12
16	223	0.00	11.30	1.33	0.90	1.58	1.19
18	298	0.00	9.50	0.61	0.30	1.00	1.64
20	1621	0.00	10.00	0.53	0.20	1.01	1.92
21	288	0.00	12.20	0.53	0.20	1.02	1.91
13	4249	0.00	20.00	1.07	0.40	2.11	1.97
14	2842	0.00	15.00	1.25	0.70	1.77	1.42
15	26	0.25	7.20	1.28	0.95	1.47	1.15
17	1907	0.00	12.00	1.05	0.56	1.61	1.54
19	194	0.00	10.00	0.98	0.65	1.18	1.21
22	76	0.25	7.70	1.39	1.10	1.28	0.92
23	34	0.25	2.30	0.78	0.70	0.56	0.72
24	34	0.25	5.70	1.80	1.45	1.53	0.85
25	343	0.00	10.00	0.64	0.35	0.97	1.51

Composites

Composites were created after capping of assay values. The assay intervals situated within the mineralization wireframe were composited to two metre lengths within each mineralized domain wireframe. Composite lengths shorter than 0.5 m were discarded.

The Z87 Zone composites average 1.96 m in length. Of the 16,189 composites, only 647 composites (approximately 4%) are lengths less than two metres.

Table 14-13, Table 14-14, and Table 14-15 present the descriptive statistics for the capped 2 m composites for gold, copper, and silver, respectively.

Table 14-13: Descriptive Statistics for 2 m Composite Values for Gold (gpt Au) – Z87 Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
1001	3750	0.00	26.00	1.56	1.12	1.75	1.12
1002	283	0.03	14.90	1.54	1.08	1.73	1.12
1003	48	0.11	10.69	1.97	1.10	2.13	1.08
11	4069	0.00	7.92	0.53	0.39	0.56	1.06
12	249	0.00	5.00	0.42	0.28	0.53	1.25
16	144	0.01	5.00	0.31	0.20	0.60	1.89
18	165	0.00	7.54	0.75	0.39	1.05	1.41
20	1054	0.00	7.00	0.62	0.40	0.74	1.19
21	174	0.00	3.61	0.52	0.39	0.51	0.97
13	2622	0.00	20.00	0.70	0.39	1.25	1.79
14	1792	0.00	15.00	0.69	0.38	1.00	1.44
15	26	0.03	1.81	0.55	0.35	0.52	0.94
17	1284	0.00	10.00	0.59	0.35	0.87	1.48
19	131	0.00	3.76	0.61	0.38	0.76	1.24
22	60	0.00	3.09	0.37	0.24	0.43	1.18
23	22	0.00	1.56	0.45	0.34	0.44	0.97
24	34	0.03	1.49	0.40	0.32	0.36	0.90
25	282	0.00	5.00	0.53	0.33	0.62	1.18

Table 14-14: Descriptive Statistics for 2 m Composite Values for Copper (%Cu) – Z87 Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
1001	3750	0.00	3.13	0.16	0.11	0.17	1.07
1002	283	0.00	1.18	0.21	0.15	0.19	0.90
1003	48	0.01	0.61	0.13	0.09	0.13	0.97
11	4069	0.00	1.23	0.06	0.04	0.07	1.15
12	249	0.00	0.53	0.09	0.07	0.08	0.91
16	144	0.00	0.54	0.09	0.07	0.08	0.88
18	165	0.00	0.18	0.04	0.02	0.04	1.01
20	1054	0.00	0.56	0.04	0.02	0.05	1.23
21	174	0.00	0.18	0.02	0.02	0.03	1.09
13	2622	0.00	2.17	0.10	0.05	0.14	1.47
14	1792	0.00	0.55	0.05	0.03	0.06	1.13
15	26	0.01	0.14	0.05	0.03	0.04	0.77
17	1284	0.00	2.54	0.07	0.03	0.15	2.02
19	131	0.00	0.33	0.04	0.02	0.05	1.27
22	60	0.01	0.36	0.08	0.07	0.07	0.82
23	22	0.01	0.16	0.05	0.04	0.04	0.79
24	34	0.00	0.14	0.03	0.02	0.03	0.88
25	282	0.00	0.60	0.03	0.02	0.05	1.47

Table 14-15: Descriptive Statistics for 2 m Composite Values for Silver (gpt Ag) – Z87 Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
1001	3750	0.00	20.00	1.44	1.09	1.65	1.14
1002	283	0.00	20.00	2.79	1.67	3.32	1.19
1003	48	0.00	4.13	0.39	0.00	0.78	1.99
11	4069	0.00	15.00	0.91	0.60	1.17	1.29
12	249	0.00	9.10	2.00	1.51	1.90	0.95
16	144	0.00	6.80	1.21	0.89	1.20	0.99
18	165	0.00	8.50	0.65	0.35	0.93	1.44
20	1054	0.00	10.00	0.63	0.35	1.01	1.60
21	174	0.00	12.20	0.60	0.35	1.13	1.89
13	2622	0.00	20.00	1.11	0.65	1.73	1.57
14	1792	0.00	15.00	1.37	0.90	1.79	1.30
15	26	0.25	7.20	1.28	0.95	1.47	1.15
17	1284	0.00	12.00	1.15	0.77	1.47	1.28
19	131	0.00	5.17	1.08	0.85	0.92	0.85
22	60	0.25	5.20	1.38	1.11	1.08	0.78
23	22	0.25	1.70	0.77	0.65	0.52	0.67
24	34	0.25	5.70	1.80	1.45	1.53	0.85
25	282	0.00	9.70	0.63	0.41	0.81	1.29

Density Assignment

Density test work, completed by Inmet, was collected from 2,721 core samples in 30 drill holes (KN-648 to KN-677). The core samples tested were generally whole core pieces ranging in length from approximately 10 cm to 20 cm. Samples were weighed in air and in water by mine personnel, and the density results were adjusted to account for water temperature. Measurements on 496 resource related samples range from 2.57 g/cm³ to 3.42 g/cm³ and average 2.86 g/cm³. The mean value was assigned to the mineralized domains.

Table 14-16 shows the densities used in the Z87 Zone.

Table 14-16: Bulk Density – Z87 Zone

	Mineralized Domains	Country/Waste Rock	Overburden
Density (g/cm3)	2.86	2.77	2.20

It is AGP's opinion, the bulk densities assigned to the mineralized domains are reasonable and acceptable. AGP recommends the collection of bulk density measurements from any future drilling program to further characterize the Z87 Zone.

During the 2019 drilling program, Troilus collected 4255 density measurements from 22 drill holes in the southern end of the Z87 Zone (Z87S area). Of these, 526 measurements are captured by the mineralized domains. These measurements were not included in the current resource estimate and are available for the next resource update.

Spatial Analysis

Spatial analysis was performed on 2 m composites from domains 11 and 12 for Z87. Experimental variograms were calculated for gold, copper, and silver and were oriented along the overall strike, dip, and across strike directions of the mineralized wireframes. The down dip and along strike gold ranges of approximately 80 m to 90 m were similar, with no significant anisotropy.

Table 14-17 to Table 14-19 presents the variography parameters for Z87 for gold, copper, and silver, respectively.

Table 14-17: Gold Variogram parameters – Z87 Zone

Sill = 1.00	Search Anisotropy	Az (°)	Dip (°)	Az (°)	X Range (m)	Y Range (m)	Z Range (m)	Variogram Type
C_0 = 0.35								
C_1 = 0.10	Az.Dip.Az.	90	60	0	25	20	4	Spherical
C_2 = 0.10	Az.Dip.Az.	90	-30	0	70	80	10	Spherical

Table 14-18: Silver Variogram parameters – Z87 Zone

Sill = 1.00	Search Anisotropy	Az (°)	Dip (°)	Az (°)	X Range (m)	Y Range (m)	Z Range (m)	Variogram Type
C_0 = 0.10								
C_1 = 0.25	Az.Dip.Az.	90	60	0	28	35	4	Spherical
C_2 = 0.65	Az.Dip.Az.	90	-30	0	90	90	18	Spherical

Table 14-19: Silver Variogram parameters – Z87 Zone

Sill = 1.00	Search Anisotropy	Az (°)	Dip (°)	Az (°)	X Range (m)	Y Range (m)	Z Range (m)	Variogram Type
C_0 = 0.10								
C_1 = 0.30	Az.Dip.Az.	90	60	0	28	70	2	Spherical
C_2 = 0.60	Az.Dip.Az.	90	-30	0	90	100	16	Spherical

14.4.3 Block Model

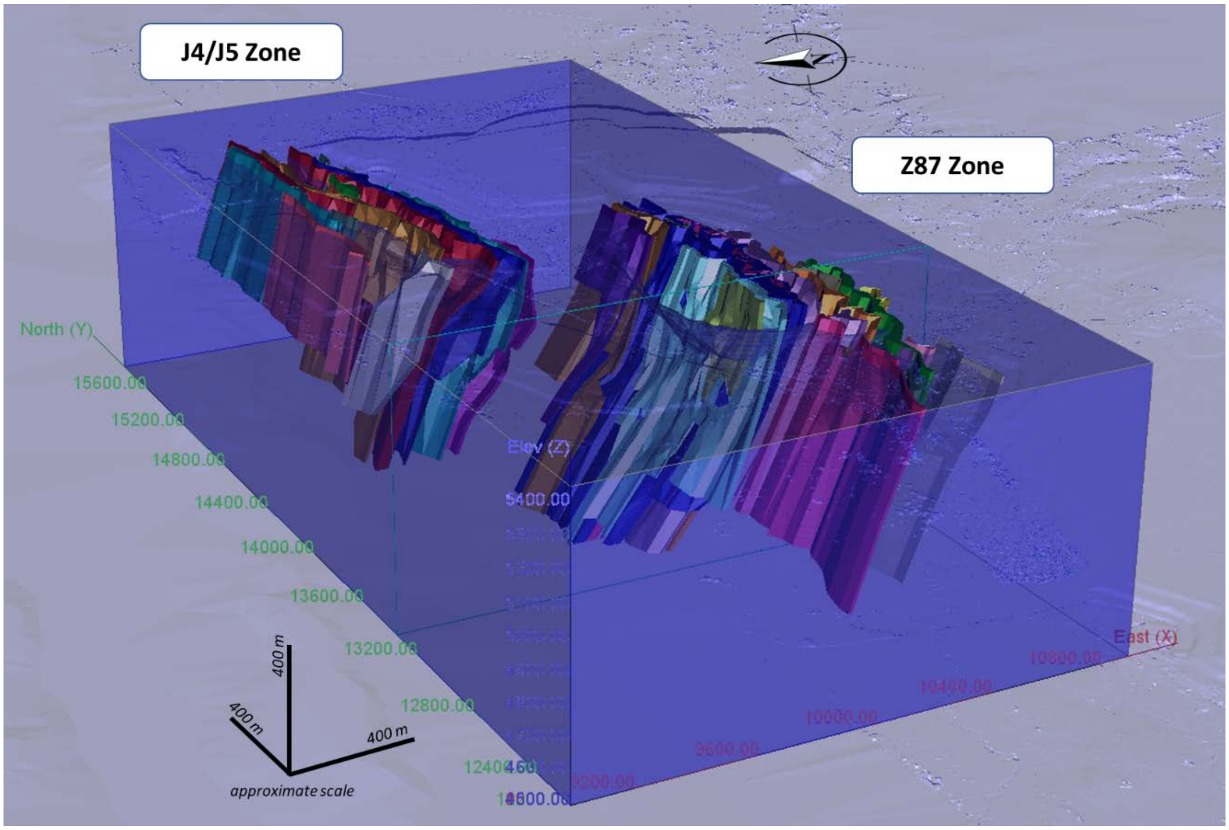
The block model for the Z87 and J4/J5 Zones was set up to cover both deposit areas. The block model was created with a block matrix of 5 m long by 5 m long by 5 m high and is not rotated. The block matrix was selected as appropriate based on the drill spacing and the block height and in consideration of an open pit and underground scenarios.

Table 14-20 summarizes the block model parameters and Figure 14-3 illustrates the block model over the interpreted mineralized domains for the Z87 Zone.

Table 14-20 Block Model Parameters – Z87 and J4/J5 Zones

	Parameters
Easting	9000 mE
Northing	12200 mN
Maximum Elevation	5455 m
Rotation Angle	No rotation°
Block Size (X, Y, Z in metres)	5m x 5m x 5m
Number of blocks in the X direction	380
Number of blocks in the Y direction	730
Number of blocks in the Z direction	191

Figure 14-3: Block Model – Z87 and J4/J5 Zones



Source : AGP (2020)

The block model is a whole block model where blocks are assigned a specific rock type code. Any block with greater than 48% within the mineralized domain wireframe was assigned that code.

Block model attributes in the block model include:

- rock type
- density
- metal grades for gold, copper, silver, and calculated gold-equivalent grades for mineralized blocks
- classification
- distance to the nearest composite
- number of composites used in estimation of block
- number of drill holes used for estimation of block
- pass number
- open pit or underground tag

Estimation/Interpolation Methods

The metal grades were interpolated in two passes using the 2 m capped composites. The metal grades were interpolated using OK interpolation method in two passes. The search ellipse ranges resemble the variogram ranges. ID2 and NN interpolations were also run for validation purposes.

Each pass required the same minimum and maximum number of composites with a maximum of three composites per drill hole, therefore, two drill holes were required to populate a block. Table 14-21 shows estimation parameters for each pass used to estimate metal grades.

Table 14-21: Estimation Parametres – Z87 Zone

Pass	Min No Composites	Max No Composites	Max Composites per Drill Holes	Min No of Drill Holes
Pass 1	4	12	3	2
Pass 2	2	12	3	1

Each pass increased the search ellipse where Pass 2 was doubled that of Pass 1. Hard boundaries were kept between all domains and blocks within each domain were estimated only be composites withing the domain wireframe. Table 14-22 shows the search ellipse parameters for the Z87 Zone

Table 14-22: Search Ellipse Parameters – Z87 Zone

Pass	Anisotropy	Azimuth (°)	Dip (°)	Azimuth (°)	Range X (m)	Range Y (m)	Range Z (m)	Search
PASS 1	Az,Dip,Az	90	60	0	70	80	18	Ellipsoidal
PASS 2	Az,Dip,Az	90	60	0	150	160	30	Ellipsoidal

Az,Dip,Az – Azimuth, Dip, Azimtuh

Block Model Validation

AGP validated the Z87 Zone resource estimate and have accepted it. Various methods to validate the block model included:

- 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

The block grades were compared with the composite grades on sections and plans and found good overall visual correlation. Occasional minor grade smearing and banding occur locally when changes in wireframe dip or strike restrict the access to composites. As the Project advances and closer spaced definition drilling becomes available, additional refinements would be possible to the mineralized domains and interpolation procedure.

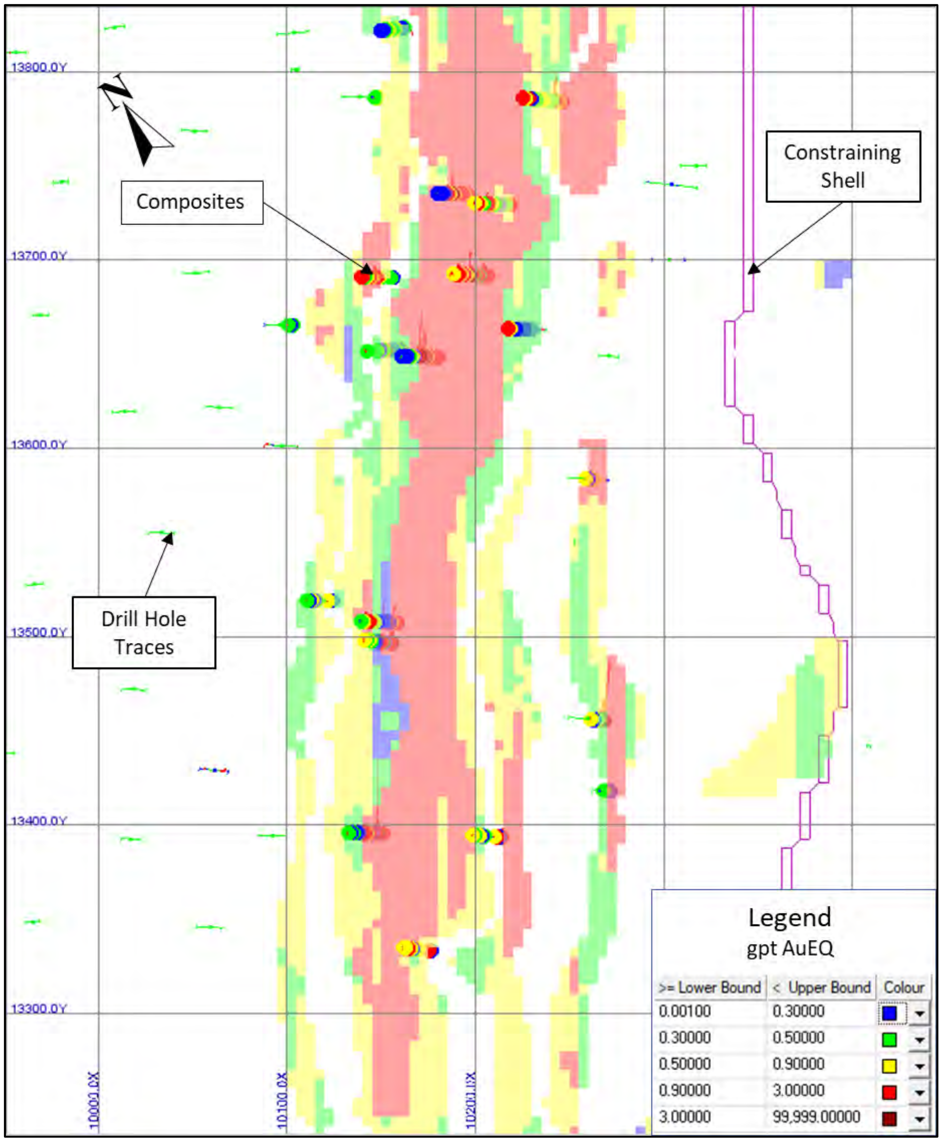
Table 14-23 presents a comparison of the gold and copper averages by lens between capped assays, composites, and estimated average block grades. The comparison between composites and interpolated block values shows a slight, normal decrease of average grades for both gold and copper. The differences in average grade of the different types of data are minimal, while occasional larger differences are observed in the case for the south extension of Z87 mineralized domains, where drill hole spacing is wider and the lenses are thinner, however, the comparison includes all the interpolated blocks, prior to classification.

Table 14-23: Comparison of Assay, Composite and Block Mean Gold Grades – Z87 Zone

Domain	Assay Au (g/t)	Comp Au (g/t)	Block Au (g/t)	Assay Cu (%)	Comp Cu (%)	Block Cu (%)	Assay Ag (g/t)	Comp Ag (g/t)	Block Ag (g/t)
1001	1.596	1.560	1.627	0.161	0.159	0.147	1.409	1.445	1.176
1002	1.630	1.538	1.434	0.214	0.207	0.170	2.868	2.789	2.298
1003	2.114	1.974	2.078	0.138	0.130	0.153	0.240	0.390	0.420
11	0.545	0.531	0.464	0.064	0.063	0.061	0.850	0.909	0.908
12	0.412	0.420	0.456	0.095	0.091	0.097	2.099	1.995	2.130
16	0.292	0.315	0.394	0.090	0.085	0.091	1.326	1.211	1.512
18	0.719	0.745	0.620	0.034	0.035	0.028	0.608	0.648	0.604
20	0.612	0.619	0.604	0.037	0.037	0.035	0.525	0.634	0.748
21	0.531	0.524	0.429	0.024	0.023	0.023	0.531	0.596	0.993
13	0.714	0.702	0.616	0.103	0.097	0.083	1.068	1.107	0.873
14	0.740	0.695	0.550	0.050	0.049	0.056	1.248	1.371	1.138
15	0.551	0.551	0.506	0.045	0.045	0.048	1.275	1.275	1.917
17	0.583	0.587	0.603	0.077	0.072	0.071	1.047	1.152	1.094
19	0.615	0.614	0.684	0.044	0.035	0.037	0.977	1.076	1.093
22	0.356	0.367	0.369	0.086	0.083	0.086	1.389	1.383	1.282
23	0.472	0.451	0.510	0.049	0.046	0.045	0.779	0.772	0.783
24	0.403	0.403	0.429	0.030	0.030	0.026	1.796	1.796	1.576
25	0.523	0.532	0.580	0.033	0.031	0.031	0.638	0.633	0.636

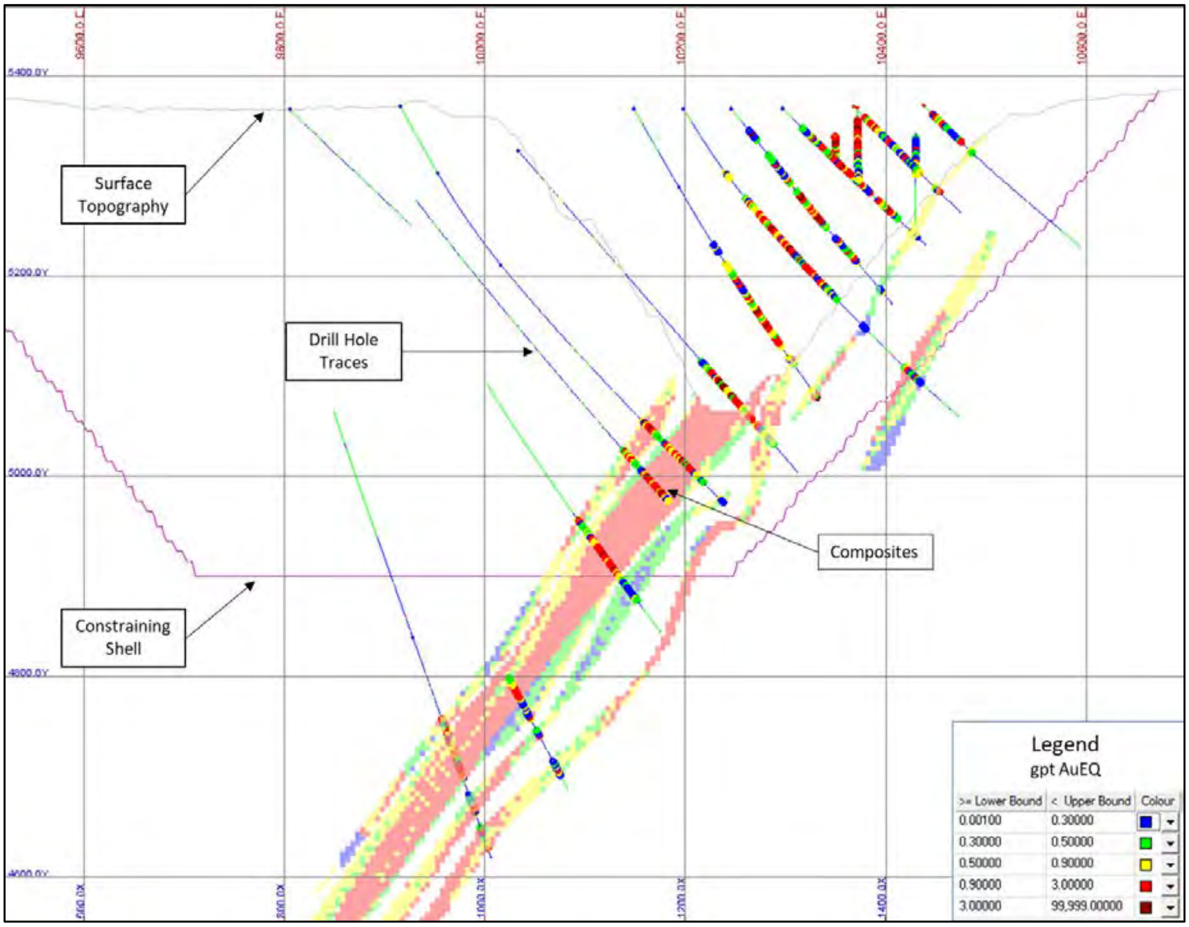
Figure 14-4 and present plan section and cross section views for Z87 (plan view elevation 5,005 and cross section 13,700N).

Figure 14-4: Z87 Zone - Plan View Elevation 5,005 m



Source : AGP (2020)

Figure 14-5: Z87 Zone – Cross Section 13,700N



Source : AGP (2020)

Swath plots by northing and by elevation reviewed in the Z87 Zone. The distribution of gold and copper composite and interpolated block grades were compared. No issues were found with the distribution of interpolated grades. Figure 14-6 and Figure 14-7 present the swath plots by northing for gold and copper.

Figure 14-6: Gold Swath Plot by Northing

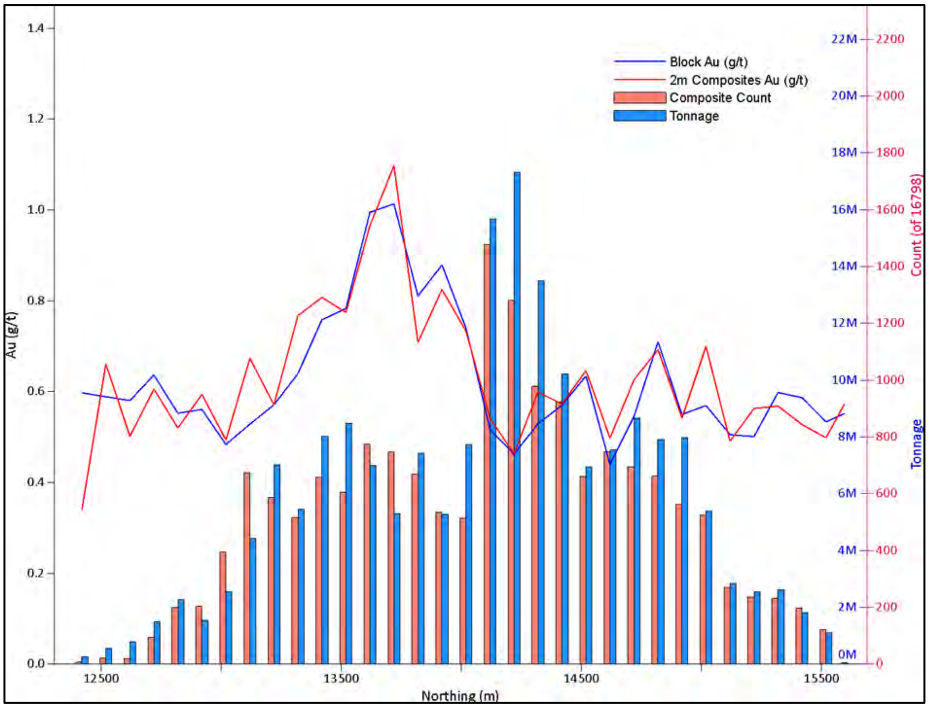
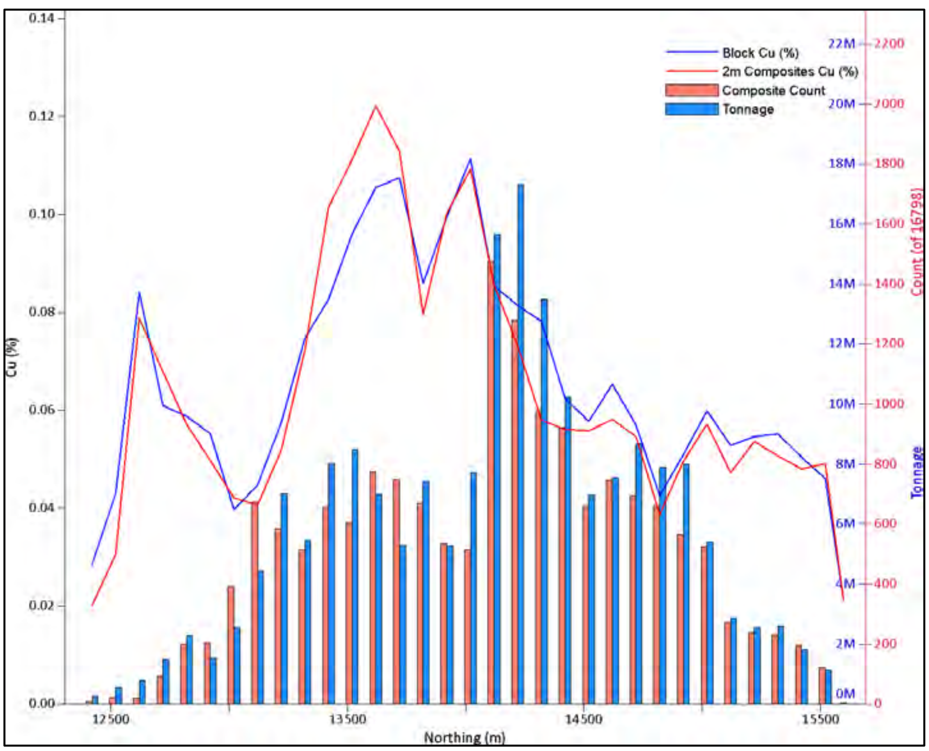


Figure 14-7: Copper Swath Plot by Northing



14.5 J4/J5 Zones

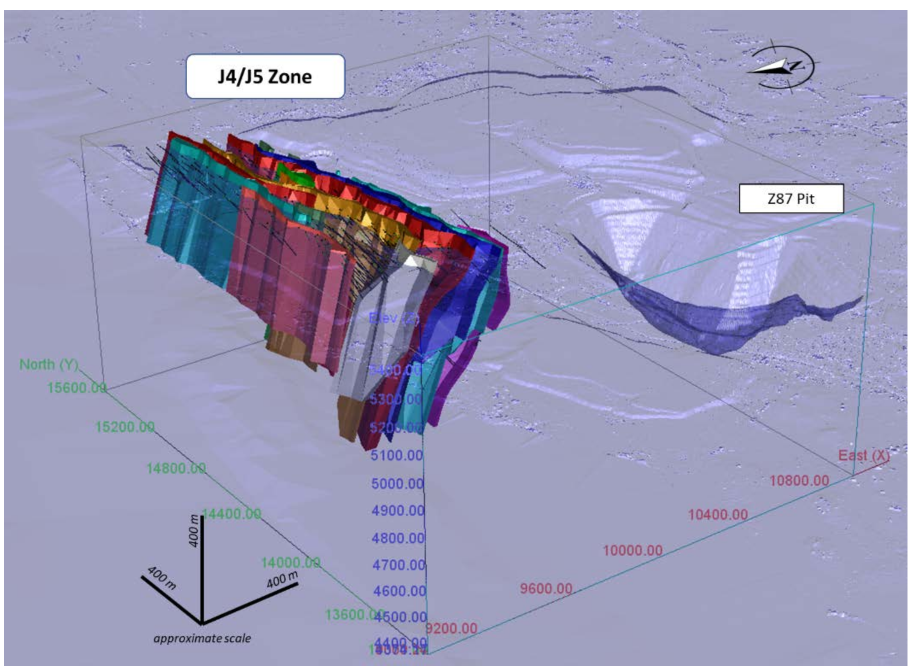
14.5.1 Geological Models

The mineralized domains at J4/J5 Zone were interpreted by Troilus personnel. The interpreted wireframes were completed using conventional polylines on vertical sections defined along 25 m spaced sections. The polylines capture a minimum nominal grade of 0.3 gpt AuEQ to a minimum width of 4 m. The polylines were 'wobbled' (GEMS function) and snapped to drill hole intercepts. The polylines were then joined together using tie lines in order to create 3D solid wireframes. The mineralized envelops were created above topography and extended approximately 600 m below surface. A total of 16 wireframes were created for the J4/J5 Zone.

AGP reviewed the J4/J5 Zone wireframes and found no errors. AGP agrees that these wireframes are suitable to estimate resources for the J4/J5 Zone. AGP notes that the wireframes exhibit some zig-zag, or sawtooth, shapes when viewed in plan view. While these shapes are not overly exaggerated, it is recommended that these wireframes be revisited to adjust along plan views in plan view. AGP anticipates these changes would not have a significant affect on the current mineral resources but may show a better representation of the mineralization.

Figure 14-8 shows the mineralized wireframes for the J4/J5 Zone. Table 14-24 lists the mineralized domains and rock type code.

Figure 14-8: Mineralized Domains – J4/J5 Zone



Source : AGP (2020)

Table 14-24: Domains – J4/J5 Zone

Zone	Domain	Rock Type
J4	40	40
	41	41
	42	42
	43	43
	44	44
	45	45
	46	46
	47	47
J5	50	50
	51	51
	52	52
	54	54
	55	55

14.5.2 Exploratory Data Analysis

Raw Assays

The drill hole database for J4/J5 Zone data, consists of 277 drill holes and 46,026 assay values for each metal: gold, copper, and silver. Any assay values reported below detection limit were assigned half the detection limit for statistical analysis and grade estimation. Any missing values were assigned a zero. Out of this total, 15,270 assay values are captured by the mineralized domains.

Table 14-25 presents the descriptive statistics for all assays in the J4/J5 Zone.

Table 14-25: Descriptive Statistics of Raw Assays within Mineralized Domains – J4/J5 Zone

	Au (gpt)	Cu (%)	Ag (gpt)	Length (m)
Count	15270	15270	15270	15270
Minimum	0.00	0.00	0.00	0.10
Maximum	94.13	3.91	206.00	5.25
Mean	0.65	0.06	0.95	1.38
Median	0.33	0.04	0.60	1.00
Std. Deviation	1.68	0.07	2.11	0.57
CV	2.60	1.22	2.22	0.41

Table 14-26 to Table 14-28 presents the descriptive statistics for raw gold, copper, and silver assays, respectively, by mineralized domain.

Table 14-26: Descriptive Statistics for Raw Gold Assays (gpt Au) – J4/J5 Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
40	319	0.00	3.12	0.34	0.23	0.36	1.09
41	3618	0.00	30.29	0.58	0.33	1.26	2.16
42	4888	0.00	94.13	0.88	0.42	2.50	2.85
43	2157	0.00	28.69	0.70	0.38	1.32	1.89
44	343	0.00	4.74	0.50	0.32	0.59	1.17
45	202	0.00	7.91	0.35	0.24	0.59	1.72
46	150	0.02	3.08	0.52	0.34	0.53	1.01
47	168	0.01	6.45	0.50	0.29	0.76	1.51
50	711	0.01	20.84	0.44	0.28	0.91	2.05
51	488	0.00	8.05	0.56	0.31	0.89	1.57
52	101	0.00	0.79	0.18	0.14	0.14	0.77
54	2036	0.00	15.40	0.41	0.24	0.82	2.02
55	89	0.02	5.88	0.42	0.25	0.73	1.73

Table 14-27: Descriptive Statistics for Raw Copper Assays (%Cu) – J4/J5 Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
40	319	0.00	0.44	0.06	0.05	0.06	0.95
41	3618	0.00	2.01	0.07	0.05	0.08	1.16
42	4888	0.00	1.37	0.05	0.03	0.05	1.10
43	2157	0.00	3.91	0.06	0.04	0.10	1.84
44	343	0.00	0.51	0.07	0.05	0.07	1.00
45	202	0.00	0.37	0.09	0.08	0.06	0.67
46	150	0.00	0.64	0.09	0.04	0.12	1.27
47	168	0.00	0.09	0.02	0.02	0.02	0.86
50	711	0.00	0.43	0.06	0.04	0.05	0.89
51	488	0.00	0.64	0.05	0.04	0.06	1.07
52	101	0.00	0.50	0.14	0.11	0.09	0.69
54	2036	0.00	0.85	0.07	0.05	0.06	0.92
55	89	0.01	0.25	0.08	0.07	0.05	0.67

Table 14-28 Descriptive Statistics for Raw Silver Assays (gpt Ag) – J4/J5 Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
40	319	0.00	6.00	0.64	0.40	0.70	1.10
41	3618	0.00	46.90	1.11	0.80	1.49	1.35
42	4888	0.00	206.00	0.91	0.60	3.15	3.45
43	2157	0.00	62.80	1.06	0.70	1.76	1.66
44	343	0.00	7.30	1.34	1.00	1.24	0.93
45	202	0.25	8.60	0.78	0.60	0.83	1.06
46	150	0.10	11.60	1.49	0.70	1.86	1.25
47	168	0.10	2.00	0.36	0.25	0.27	0.76
50	711	0.00	12.20	0.87	0.60	1.01	1.16
51	488	0.00	4.60	0.53	0.30	0.59	1.11
52	101	0.00	2.40	0.62	0.55	0.44	0.71
54	2036	0.00	15.40	0.82	0.60	0.94	1.15
55	89	0.10	3.70	0.76	0.25	0.77	1.02

Capping Analysis

Capping analysis was carried out on each mineralized domain for gold, copper, and silver. Capping was applied to gold and silver assay values in several mineralized domains. Not all domains required capping levels. No capping was applied to copper assay values. AGP reviewed the capping levels by domain using histogram and disintegration plots and found the capping levels to be reasonable and adequate.

Table 14-29 presents the capping levels for gold and silver, by domain, for the J4/J5 Zone. Table 14-30 and Table 14-31 present the descriptive statistics for capped gold and silver assay values, respectively.

Table 14-29 Capping Levels – J4/J5 Zone

Domain	Au (gpt)	% Loss	Cu (%)	% Loss	Ag (gpt)	% Loss
40	2.00 (2)	1.1	-		3.00 (5)	2.9
41	8.00 (14)	5.2	-		9.00 (12)	2.5
42	14.00 (9)	7.1	-		9.00 (4)	5.8
43	8.00 (9)	4.2	-		9.00 (6)	2.9
44	-		-		6.00 (3)	0.7
45	2.00 (2)	18.7	-		3.00 (2)	4.8
46	-		-		6.00 (6)	5.1
47	3.00 (2)	6.2	-		1.00 (5)	2.5
50	4.00 (2)	5.6	-		6.00 (4)	1.3
51	-		-		-	
52	-		-		-	
54	7.00 (8)	3.4	-		8.00 (3)	0.7
55	3.00 (1)	7.7	-		-	

(X) – number of assays affected

Table 14-30 and Table 14-31 presents the descriptive statistics for capped gold and silver assay values, respectively. There is no change to copper values as no capping was applied.

Table 14-30 Descriptive Statistics for Capped Gold Assays (gpt Au) – J4/J5 Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
40	319	0.00	2.00	0.33	0.23	0.34	1.01
41	3618	0.00	8.00	0.55	0.33	0.81	1.47
42	4888	0.00	14.00	0.81	0.42	1.37	1.68
43	2157	0.00	8.00	0.67	0.38	0.96	1.44
44	343	0.00	4.74	0.50	0.32	0.59	1.17
45	202	0.00	2.00	0.31	0.24	0.27	0.85
46	150	0.02	3.08	0.52	0.34	0.53	1.01
47	168	0.01	3.00	0.47	0.29	0.58	1.22
50	711	0.01	4.00	0.42	0.28	0.50	1.19
51	488	0.00	8.05	0.56	0.31	0.89	1.57
52	101	0.00	0.79	0.18	0.14	0.14	0.77
54	2036	0.00	7.00	0.39	0.24	0.65	1.65
55	89	0.02	3.00	0.39	0.25	0.50	1.29

Table 14-31: Descriptive Statistics for Capped Silver Assays (gpt Ag) – J4/J5 Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
40	319	0.00	6.00	0.64	0.40	0.70	1.10
41	3618	0.00	9.00	1.08	0.80	1.10	1.02
42	4888	0.00	9.00	0.86	0.60	0.86	1.00
43	2157	0.00	9.00	1.03	0.70	1.12	1.09
44	343	0.00	6.00	1.33	1.00	1.20	0.90
45	202	0.25	2.70	0.74	0.60	0.58	0.78
46	150	0.10	6.00	1.41	0.70	1.58	1.11
47	168	0.10	1.20	0.35	0.25	0.25	0.71
50	711	0.00	6.00	0.86	0.60	0.92	1.07
51	488	0.00	4.60	0.53	0.30	0.59	1.11
52	101	0.00	2.40	0.62	0.55	0.44	0.71
54	2036	0.00	8.00	0.81	0.60	0.88	1.08
55	89	0.10	3.70	0.76	0.25	0.77	1.02

Composites

Composites were created after capping of assay values. The assay intervals situated within the mineralization wireframe were composited to two metre lengths within each mineralized domain wireframe. Composite lengths shorter than 0.5 m were discarded.

The J4/J5 Zone composites average 1.99 m in length. Of the 32,578 composites, only 273 composites (approximately 1%) are less than two metre lengths.

Table 14-32 to Table 14-34 present the descriptive statistics for the capped 2 m composites for gold, copper, and silver, respectively.

Table 14-32: Descriptive Statistics for 2 m Composite Values for Gold (gpt Au) – J4/J5 Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
40	217	0.00	1.82	0.33	0.26	0.28	0.85
41	2560	0.00	8.00	0.54	0.35	0.67	1.24
42	3261	0.00	14.00	0.79	0.47	1.11	1.39
43	1611	0.00	8.00	0.65	0.41	0.79	1.22
44	289	0.00	3.81	0.52	0.33	0.56	1.07
45	171	0.00	2.00	0.31	0.25	0.23	0.74
46	140	0.03	3.08	0.53	0.36	0.52	0.98
47	93	0.02	2.27	0.48	0.33	0.46	0.96
50	423	0.01	3.84	0.41	0.30	0.40	0.98
51	313	0.00	7.33	0.55	0.32	0.72	1.32
52	79	0.00	0.79	0.18	0.16	0.14	0.74
54	1496	0.00	7.00	0.40	0.26	0.62	1.54
55	68	0.03	2.58	0.37	0.25	0.41	1.09

Table 14-33: Descriptive Statistics for 2 m Composite Values for Copper (%Cu) – J4/J5 Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
40	0.00	0.33	0.06	0.05	0.05	0.78	0.00
41	0.00	2.01	0.06	0.05	0.07	1.10	0.00
42	0.00	0.70	0.04	0.04	0.04	0.96	0.00
43	0.00	0.91	0.05	0.04	0.06	1.05	0.00
44	0.00	0.51	0.07	0.05	0.07	0.90	0.00
45	0.00	0.37	0.10	0.09	0.06	0.61	0.00
46	0.00	0.64	0.10	0.05	0.12	1.21	0.00
47	0.00	0.07	0.02	0.02	0.02	0.73	0.00
50	0.00	0.27	0.06	0.05	0.04	0.76	0.00
51	0.00	0.64	0.06	0.04	0.06	1.09	0.00
52	0.01	0.50	0.14	0.12	0.09	0.68	0.01
54	0.00	0.85	0.07	0.05	0.06	0.89	0.00
55	0.01	0.25	0.08	0.07	0.04	0.57	0.01

Table 14-34: Descriptive Statistics for 2 m Composite Values for Silver (gpt Ag) – J4/J5 Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
40	0.00	2.80	0.67	0.50	0.54	0.81	0.00
41	0.00	9.00	1.10	0.09	0.94	0.85	0.00
42	0.00	8.05	0.94	0.75	0.80	0.84	0.00
43	0.00	9.00	1.10	0.09	1.07	0.98	0.00
44	0.00	6.00	1.32	1.10	1.09	0.82	0.00
45	0.25	2.50	0.76	0.60	0.55	0.73	0.25
46	0.15	6.00	1.48	0.88	1.58	1.07	0.15
47	0.10	1.00	0.35	0.30	0.21	0.59	0.10
50	0.00	5.20	0.88	0.62	0.84	0.95	0.00
51	0.00	3.55	0.58	0.36	0.57	0.98	0.00
52	0.06	2.40	0.64	0.60	0.42	0.65	0.06
54	0.00	8.00	0.84	0.60	0.85	1.01	0.00
55	0.10	3.40	0.82	0.50	0.77	0.94	0.10

Density Assignment

Density test work, completed by Troilus, was collected from 13,409 core samples in 46 drill holes from the 2019 drill programs. Of this total, 3,356 measurements were used for the mineralized domains. Measurements were carried out by ALS by weight in air and weight in water method. There is a slight variation in the bulk densities between the J4 and J5 domains therefore: 2.77 g/cm^3 was assigned to the mineralized domains at J4 domains and 2.80 g/cm^3 was assigned to J5 domains.

Table 14-35 shows the densities used in the J4/J5Zone

Table 14-35: Assigned Densities – J4/J5 Zone

	Mineralized Domains	Country/Waste Rock	Overburden
J4 Domains	2.77	2.77	2.20
J5 Domains	2.80	2.77	2.20

Table 14-36 presents the descriptive statistics for the density values assigned to the mineralized domains in J4/J5 Zone.

Table 14-36: Bulk Density – J4/J5 Zone

	J4 Domains Density (g/cm^3)	J5 Domains Density (g/cm^3)	Country Rock Density (g/cm^3)
Count	2376	980	10053
Minimum	2.18	2.17	2.10
Maximum	3.03	3.17	3.63
Mean	2.77	2.80	2.77
Median	2.78	2.77	2.77
StDev	0.06	0.10	0.07
CV	0.02	0.03	0.03

It is AGP's opinion, the bulk densities assigned to the mineralized domains is reasonable and acceptable. AGP recommends the continued collection of bulk density measurements from any drilling program to continue to characterize the J4/J5 Zone.

Spatial Analysis

Spatial analysis was performed on 2 m composites from domain 42. Experimental variograms were calculated for gold, copper, and silver and were oriented along the overall strike, dip, and across strike directions of the mineralized wireframes. The down dip and along strike gold ranges of approximately 80 m to 90 m were similar, with no significant anisotropy. Along strike, ranges for copper and silver were approximately twice the down dip ranges.

Table 14-37 to Table 14-39 presents the variography parameters for J4/J5 for gold, copper, and silver, respectively.

Table 14-37: Gold Variogram parameters – J4/J5 Zone

Sill = 1.00	Search Anisotropy	Az (°)	Dip (°)	Az (°)	X Range (m)	Y Range (m)	Z Range (m)	Variogram Type
C_0 = 0.35								
C_1 = 0.40	Az.Dip.Az.	90	60	0	35	55	4	Spherical
C_2 = 0.25	Az.Dip.Az.	90	-30	0	85	85	10	Spherical

Table 14-38: Silver Variogram parameters – J4/J5 Zone

Sill = 1.00	Search Anisotropy	Az (°)	Dip (°)	Az (°)	X Range (m)	Y Range (m)	Z Range (m)	Variogram Type
C_0 = 0.30								
C_1 = 0.70	Az.Dip.Az.	90	60	0	177	100	8	Spherical

Table 14-39: Silver Variogram parameters – J4/J5 Zone

Sill = 1.00	Search Anisotropy	Az (°)	Dip (°)	Az (°)	X Range (m)	Y Range (m)	Z Range (m)	Variogram Type
C_0 = 0.30								
C_1 = 0.20	Az.Dip.Az.	90	60	0	36	61	4	Spherical
C_2 = 0.50	Az.Dip.Az.	90	-30	0	205	105	9	Spherical

14.5.3 Block Model

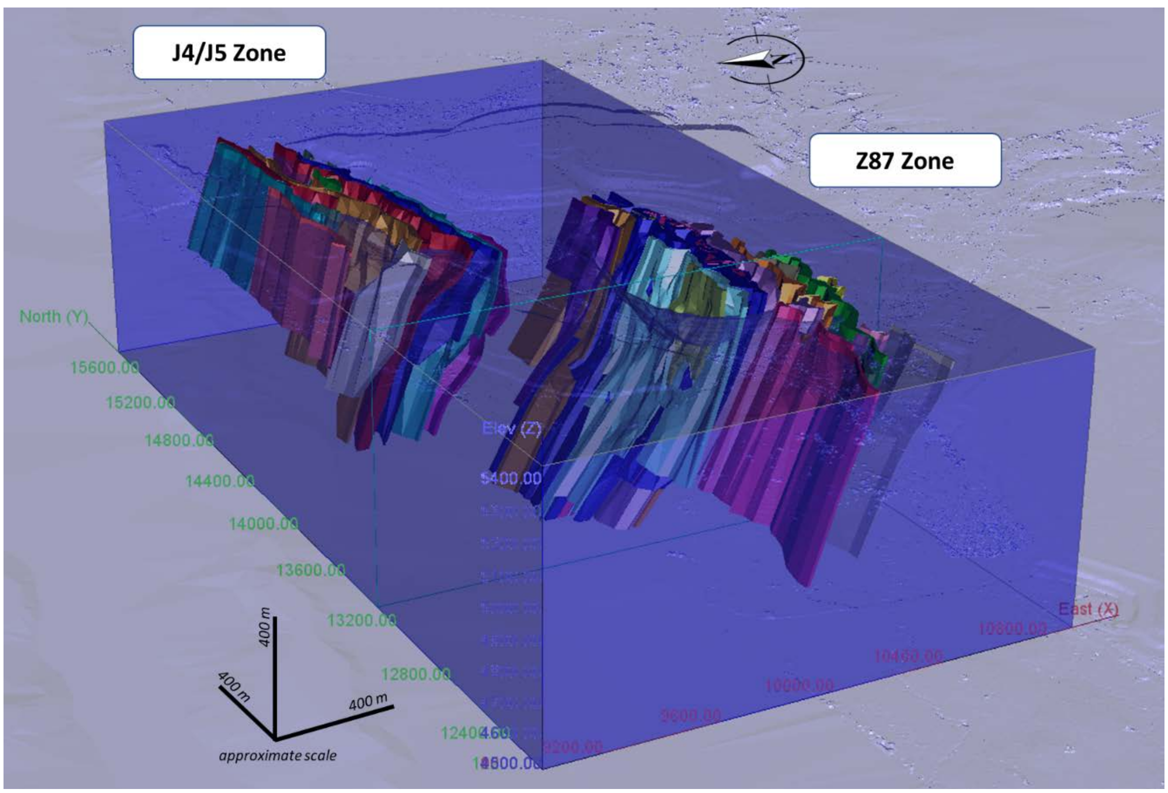
The block model for the Z87 and J4/J5 Zones was set up to cover both deposit areas. The block model was created with a block matrix of 5 m long by 5 m long by 5 m high and is not rotated. The block matrix was selected as appropriate based on the drill spacing and the block height and in consideration of an open pit and underground scenarios.

Table 14-40 summarizes the block model parameters and Figure 14-9 illustrates the block model over the interpreted mineralized domains for the Z87 Zone.

Table 14-40: Block Model Parameters – Z87 and J4/J5 Zones

	Parameters
Easting	9000 mE
Northing	12200 mN
Maximum Elevation	5455 m
Rotation Angle	No rotation°
Block Size (X, Y, Z in metres)	5m x 5m x 5m
Number of blocks in the X direction	380
Number of blocks in the Y direction	730
Number of blocks in the Z direction	191

Figure 14-9: Block Model – Z87 and J4/J5 Zones



Source : AGP (2020)

The block model is a whole block model where blocks are assigned a specific rock type code. Any block with greater than 48% within the mineralized domain wireframe was assigned that code. The volume of the coded blocks was compared to the analytical volume and was found to be within XXX%.

Block model attributes in the block model include:

- rock type
- density
- metal grades for gold, copper, silver, and calculated gold-equivalent grades for mineralized blocks
- classification
- distance to the nearest composite
- number of composites used in estimation of block
- number of drill holes used for estimation of block
- pass number
- open pit or underground tag

Estimation/Interpolation Methods

The metal grades were interpolated in two passes using the 2 m capped composites. The metal grades were interpolated using OK interpolation method in two passes. The search ellipse ranges resemble the variogram ranges. ID2 and NN interpolations were also run for validation purposes.

Each pass required the same minimum and maximum number of composites with a maximum of three composites per drill hole, therefore, two drill holes were required to populate a block. Table 14-41 shows estimation parameters for each pass used to estimate metal grades.

Table 14-41: Estimation Parametres – J4/J5 Zone

Pass	Min No Composites	Max No Composites	Max Composites per Drill Holes	Min No of Drill Holes
Pass 1	4	12	3	2
Pass 2	2	12	3	1

Each pass increased the search ellipse where Pass 2 was doubled that of Pass 1. Hard boundaries were kept between all domains and blocks within each domain were estimated only be composites withing the domain wireframe. shows the search ellipse parameters for the J4/J5 zone.

Table 14-42: Search Ellipse Parameters – J4/J5 Zone

Pass	Anisotropy	Azimuth (°)	Dip (°)	Azimuth (°)	Range X (m)	Range Y (m)	Range Z (m)	Search
PASS 1	Az,Dip,Az	90	70	0	85	85	10	Ellipsoidal
PASS 2	Az,Dip,Az	90	70	0	150	160	30	Ellipsoidal

Az,Dip,Az – Azimuth, Dip, Azimtuh

Block Model Validation

AGP validated the J4/J5 Zone resource estimate and have accepted it. The various methods used to validate the block model included:

- visual inspection and comparison of block grades with composite and assay grades
- statistical comparison of resource assay and block grade distributions
- inspection of swath plots with composites and block grades elevations and northings

The block grades were compared with the composite grades on sections and plans and found good overall visual correlation. Occasional minor grade smearing and banding occur locally when changes in wireframe dip or strike restrict the access to composites. As the Project advances and closer spaced definition drilling becomes available, additional refinements would be possible to the mineralized domains and interpolation procedure.

Table 14-43 presents a comparison of the gold and copper averages by lens between capped assays, composites, and estimated average block grades. The comparison between composites and interpolated block values shows a slight, normal decrease of average grades for both gold and copper. For J4-J5, the differences in average grade of the different types of data are minimal, while for Z87 occasional larger differences are observed as is the case for the South extension mineralized lenses, where drill hole spacing is wider and the lenses are thinner, however, the comparison includes all the interpolated blocks, prior to classification.

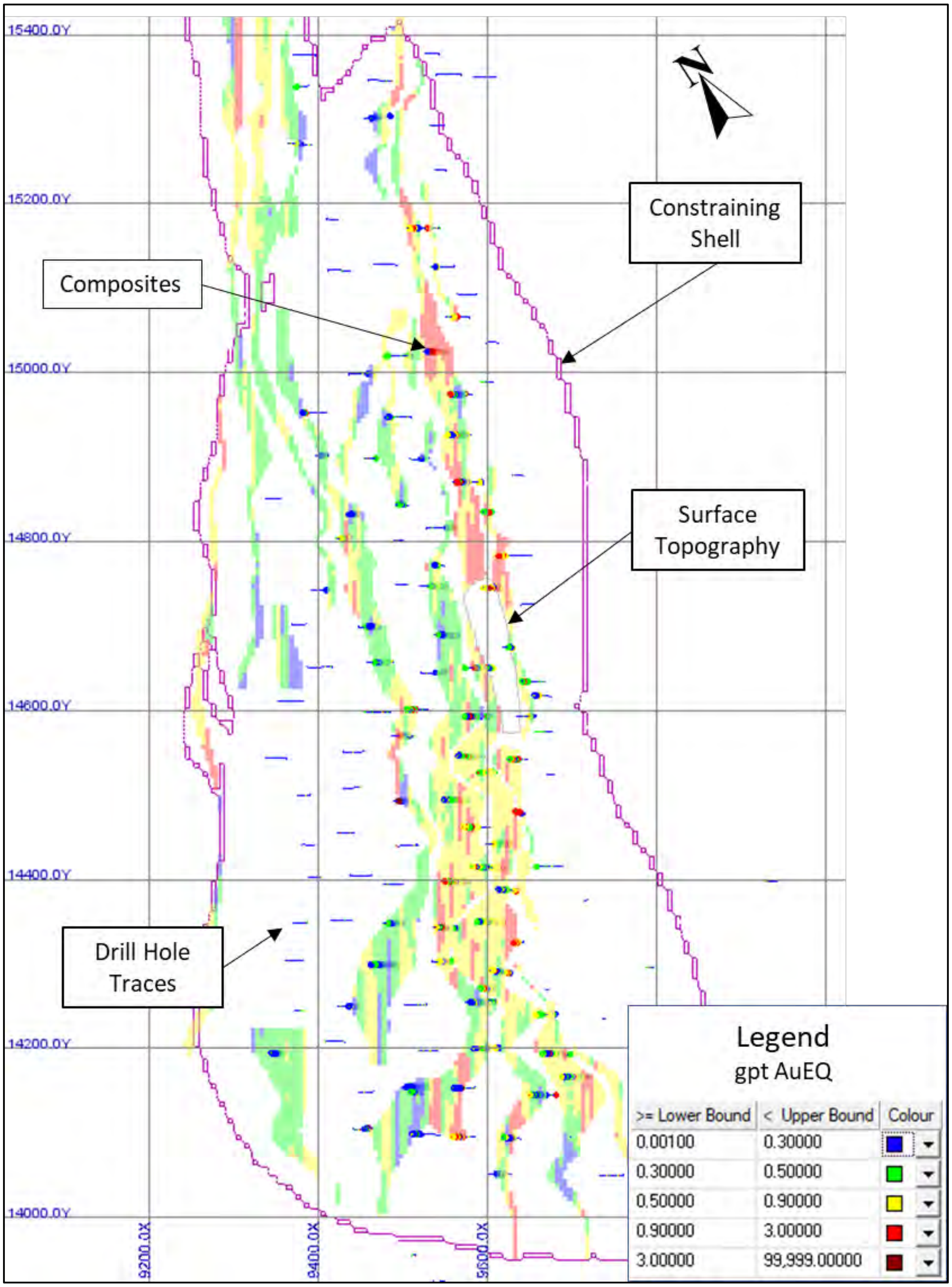
Table 14-43: Comparison of Assay, Composite and Block Mean Gold Grades – Z87 Zone

Domain	Assay Au (g/t)	Comp Au (g/t)	Block Au (g/t)	Assay Cu (%)	Comp Cu (%)	Block Cu (%)	Assay Ag (g/t)	Comp Ag (g/t)	Block Ag (g/t)
40	0.329	0.330	0.350	0.063	0.064	0.065	0.616	0.674	0.688
41	0.551	0.543	0.500	0.065	0.063	0.069	1.078	1.102	1.053
42	0.811	0.792	0.741	0.046	0.040	0.048	0.861	0.943	0.894
43	0.669	0.648	0.579	0.054	0.053	0.056	1.027	1.096	1.100
44	0.504	0.520	0.552	0.072	0.072	0.084	1.333	1.321	1.382
45	0.313	0.314	0.329	0.092	0.098	0.090	0.742	0.761	0.790
46	0.520	0.531	0.577	0.093	0.098	0.101	1.412	1.483	1.483
47	0.472	0.482	0.478	0.021	0.020	0.018	0.352	0.352	0.335
50	0.418	0.411	0.423	0.055	0.055	0.058	0.861	0.881	0.855
51	0.564	0.547	0.506	0.053	0.055	0.055	0.526	0.584	0.490
52	0.177	0.182	0.165	0.135	0.139	0.129	0.617	0.641	0.651
54	0.407	0.403	0.426	0.066	0.065	0.072	0.815	0.836	0.876
55	0.421	0.374	0.352	0.078	0.078	0.076	0.755	0.815	0.802

Notes: Comp - composite

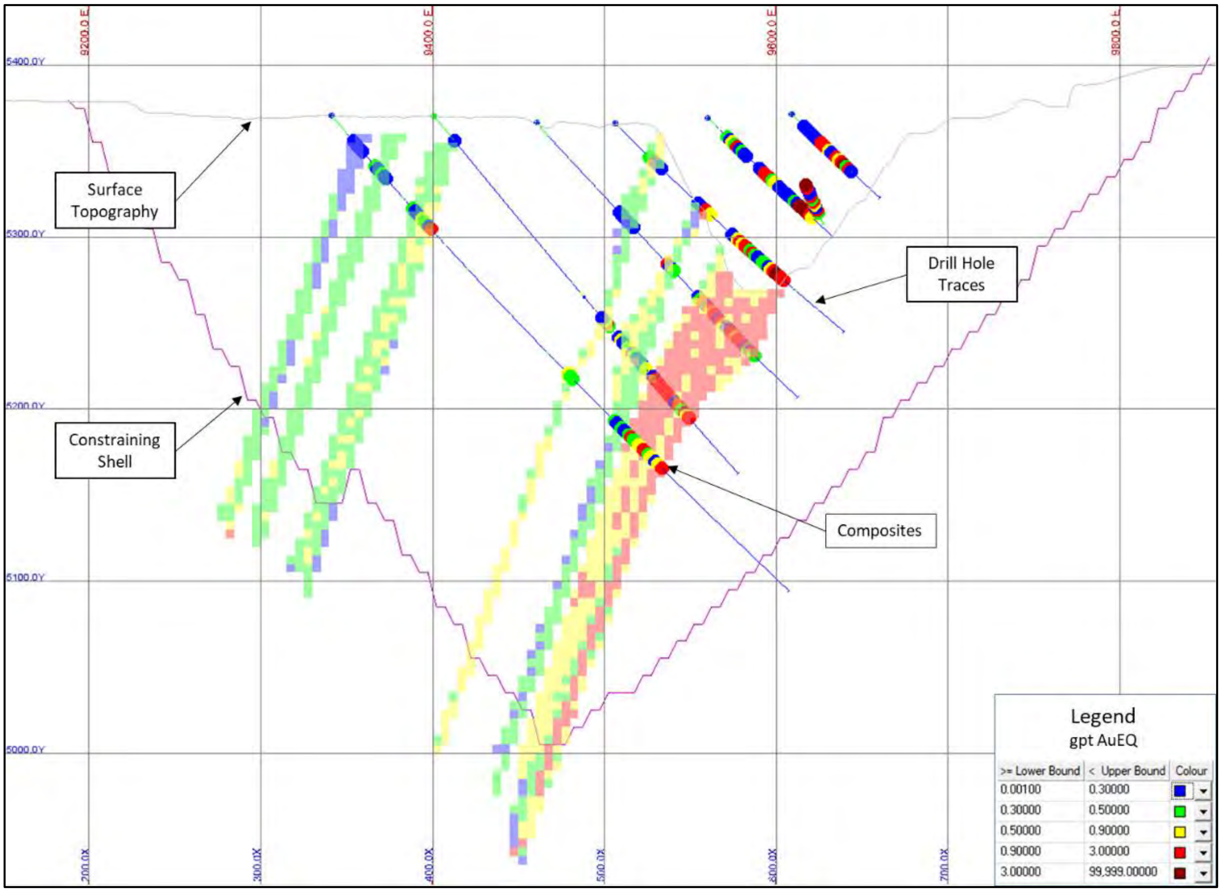
Figure 14-10 and Figure 14-11 plan section and cross section views for J4/J5 Zone (plan view elevation 5,205 and cross section 15,025N), respectively.

Figure 14-10: J4-J5 Plan View Elevation 5,205 N



Source : AGP (2020)

Figure 14-11: J4-J5 Vertical Section 15,025 N



Source : AGP (2020)

Swath plots by northing and by elevation reviewed in the Z87 Zone. The distribution of gold and copper composite and interpolated block grades were compared. No issues were found with the distribution of interpolated grades. Figure 14-12 and Figure 14-13 present the swath plots by northing for Au and Cu for all the classified blocks.

Figure 14-12: Gold Swath Plot by Northing

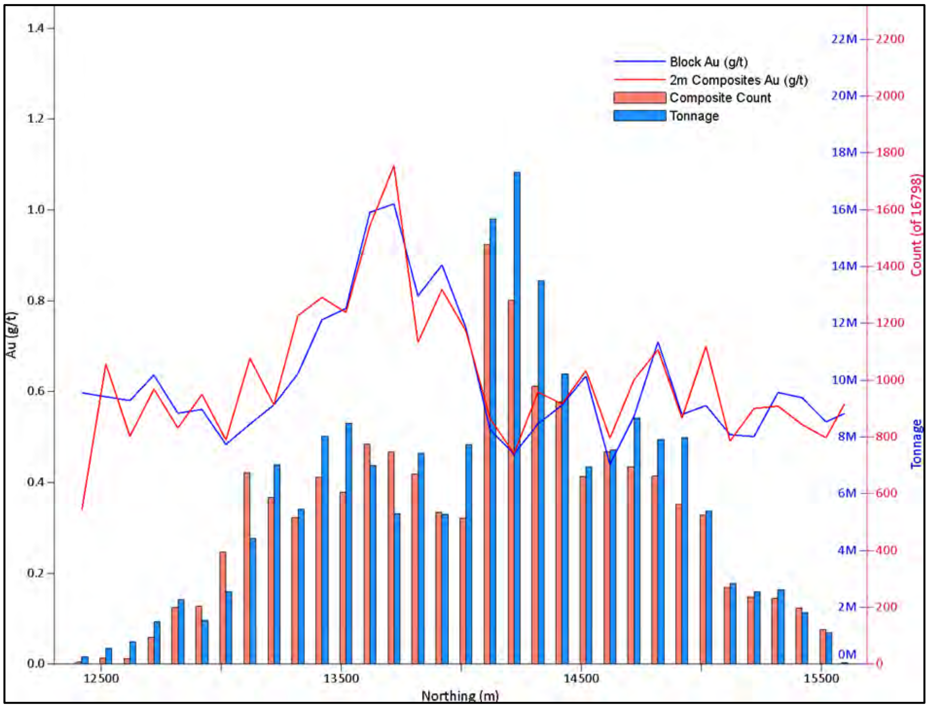
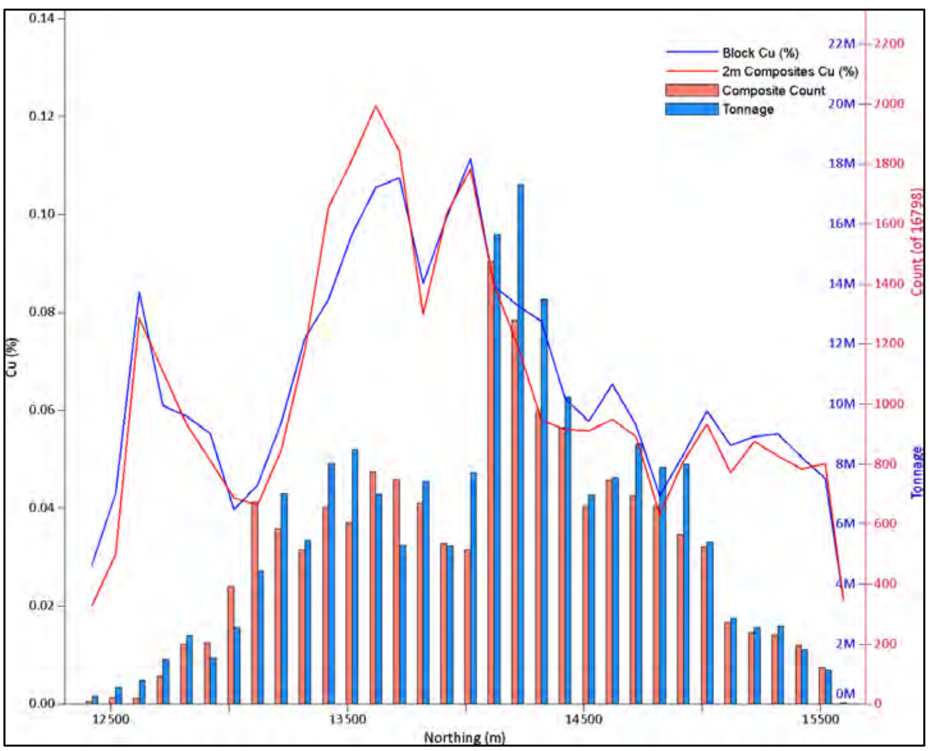


Figure 14-13: Copper Swath Plot by Northing



14.6 SW Zone

14.6.1 Geological Models

The mineralized domains at SW Zone were interpreted by Troilus personnel. The interpreted wireframes were completed using a combination of polyline rings on vertical sections and plan sections. The polylines capture a minimum nominal grade of 0.3 gpt AuEQ to a minimum width of 4 m. The polylines were then joined together using tie lines in order to create 3D solid wireframes. The mineralized envelops were created above topography and clipped to the overburden topography and extend approximately 400 m below surface. A total of eight wireframe domains were created for the SW Zone.

AGP reviewed the SW Zone wireframe domains and found no errors and agrees that these wireframes are suitable to estimate resources.

Figure 14-14 shows the mineralized domain wireframes for the SW Zone and the existing drilling. Table 14-44 lists the mineralized domains and subdomains.

Figure 14-14: Mineralized Domains – SW Zone

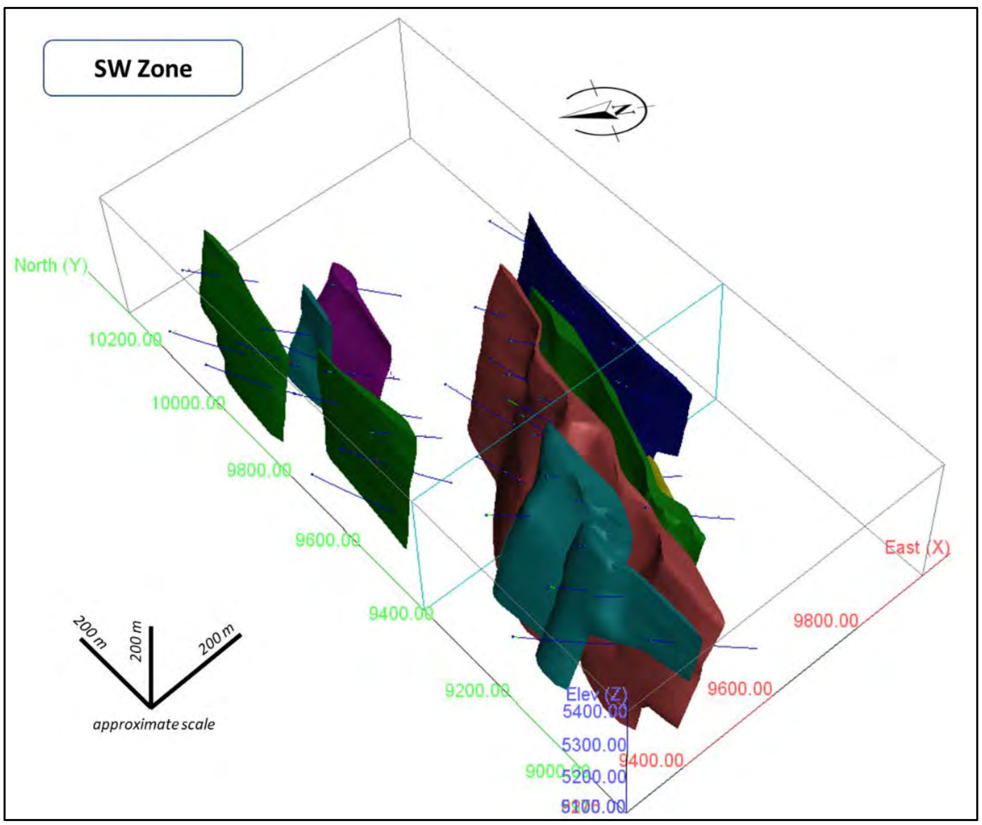


Table 14-44: Domains – SW Zone

Domain	Rock Type	Subdomain	Subdomain Type
201	201		
202	202		
203	203		
204	204		
205	205	205NT	1
		205N	2
		205S	3
206	206		
207	207		
208	208		

14.6.2 Exploratory Data Analysis

Raw Assays

The drill hole database for the SW Zone data, consists of 63 drill holes and 1,362 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 a zero.

Table 14-45 presents the descriptive statistics of the drill holes in the in the SW Zone within the mineralized domains.

Table 14-45: Descriptive Statistics on all raw assays – SW Zone

	Au (gpt)	Cu (%)	Ag (gpt)	Length (m)
Count	1362	1362	1362	1362
Minimum	0	0	0	0.21
Maximum	18.11	2.11	60.70	2.00
Mean	0.65	0.07	1.08	1.07
Median	0.30	0.03	0.25	1.00
Std. Deviation	1.23	0.12	3.06	0.281
CV	1.90	1.91	2.84	0.26

Table 14-46 to Table 14-48 presents the descriptive statistics for each metal by mineralized domains in the SW Zone.

Table 14-46: Descriptive Statistics for Gold by Mineralized Domain – SW Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
201	368	0.007	18.11	0.91	0.45	1.47	1.62
202	384	0.003	14.15	0.59	0.26	1.29	2.21
203	69	0.003	11.65	0.70	0.31	1.49	2.13
204	190	0	15.10	0.46	0.22	1.20	2.60
205	198	0.005	5.04	0.46	0.28	0.59	1.27
206	30	0.017	3.63	0.81	0.41	0.93	1.14
207	32	0.02	2.23	0.57	0.34	0.58	1.03
208	91	0.018	4.27	0.58	0.32	0.77	1.33

Table 14-47 Descriptive Statistics for Copper by Mineralized Domain – SW Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
201	368	0	2.11	0.12	0.04	0.21	1.79
202	384	0	0.55	0.04	0.02	0.06	1.43
203	69	0	0.24	0.04	0.02	0.05	1.30
204	190	0	0.42	0.08	0.06	0.07	0.92
205	198	0	0.32	0.05	0.03	0.06	1.12
206	30	0.001	0.54	0.04	0.00	0.10	2.88
207	32	0.001	0.29	0.02	0.00	0.06	2.50
208	91	0.001	0.28	0.01	0.00	0.04	2.88

Table 14-48: Descriptive Statistics for Silver by Mineralized Domain – SW Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
201	368	0.10	28.30	1.39	0.25	2.84	2.03
202	384	0.02	6.60	0.55	0.25	0.75	1.37
203	69	0.25	5.60	0.64	0.25	0.93	1.45
204	190	0	48.40	1.46	0.70	3.77	2.59
205	198	0.25	3.60	0.52	0.25	0.52	0.99
206	30	0.20	12.90	1.38	0.30	2.75	1.99
207	32	0.20	7.10	1.01	0.25	1.52	1.50
208	91	0.20	60.70	2.68	0.30	8.14	3.04

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 to gold and silver assay values in several mineralized domains. Not all domains required capping levels. No capping was applied to copper assay values. AGP reviewed the capping levels by domain using histogram and disintegration plots and found the capping levels to be reasonable and adequate.

Historically, all high grade gold resource assays at Z87 have been capped to 6.0 g/t Au prior to compositing. High grade copper assays are rare and copper assays have not historically been capped at Troilus. Reconciliation work in 2003 and 2004 indicated that the 6.0 g/t Au capping level was appropriate, however, RPA considers the 6.0 g/t Au capping level to be conservative for higher grade areas such as the deeper parts of Z87. Accordingly, a gold and silver assay capping strategy by mineralized lens was used for the current estimate. No capping was applied to copper assays. Gold and silver assays were capped before compositing. Table 14-49 presents the selected gold and silver capping levels by domain. Descriptive statistics for capped gold and silver assays are presented in Table 14-50, Table 14-51, and Table 14-52, respectively

Table 14-49: Capping Levels – SW Zone

Domain	Au (gpt)	% Loss	Cu (%)	% Loss	Ag (gpt)	% Loss
201	8.00 (1)	2.4	1.00 (2)	4.1	14.00 (3)	4.7
202	8.00 (3)	6.2	-	-	-	-
203	4.00 (1)	16.0	-	-	-	-
204	5.00 (1)	12.0	-	-	8.00 (2)	18.0
205	3.00 (1)	2.2	-	-	-	-
206	-	-	-	-	3.00 (3)	41.0
207	-	-	-	-	3.00 (2)	18.0
208	-	-	-	-	10.00 (4)	41.0

(X) – number of assays capped

Table 14-50: Descriptive Statistics for Capped Gold Assays by Mineralized Domain – SW Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
201	368	0.007	8.00	0.88	0.45	1.22	1.39
202	384	0.003	8.00	0.55	0.26	0.97	1.76
203	69	0.003	4.00	0.59	0.31	0.78	1.33
204	190	0	5.00	0.41	0.22	0.64	1.57
205	198	0.005	3.00	0.45	0.28	0.52	1.16
206	30	0.017	3.63	0.81	0.41	0.93	1.14
207	32	0.02	2.23	0.57	0.34	0.58	1.03
208	91	0.018	4.27	0.58	0.32	0.77	1.33

Table 14-51: Descriptive Statistics for Capped Copper Assays by Mineralized Domain – SW Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
201	368	0	1.00	0.11	0.04	0.17	1.56
202	384	0	0.55	0.04	0.02	0.06	1.43
203	69	0	0.24	0.04	0.02	0.05	1.30
204	190	0	0.42	0.08	0.06	0.07	0.92
205	198	0	0.32	0.05	0.03	0.06	1.12
206	30	0.001	0.54	0.04	0.00	0.10	2.88
207	32	0.001	0.29	0.02	0.00	0.06	2.50
208	91	0.001	0.28	0.01	0.00	0.04	2.88

Table 14-52: Descriptive Statistics for Capped Silver Assays by Mineralized Domain – SW Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
201	368	0.1	14.00	1.33	0.25	2.37	1.78
202	384	0.2	6.60	0.55	0.25	0.75	1.37
203	69	0.25	5.60	0.64	0.25	0.93	1.45
204	190	0	8.00	1.20	0.70	1.35	1.12
205	198	0.25	3.60	0.52	0.25	0.52	0.99
206	30	0.2	3.00	0.82	0.30	0.94	1.16
207	32	0.2	3.00	0.83	0.25	0.94	1.13
208	91	0.2	10.00	1.58	0.30	2.65	1.67

Composites

The assays situated within the mineralization wireframe were composited to two metre lengths starting at domain boundary. Composites were adjusted across the intersection of the domain.

Table 14-53 to Table 14-55 show the descriptive statistics for the 2 m capped composite values for the SW Zone by domain.

Table 14-53: Descriptive Statistics for 2m Capped Composites

Domain	Count	Min	Max	Mean	StDev	CV
201	195	0.011	8.00	0.86	0.52	1.02
202	200	0.009	6.99	0.55	0.36	0.81
203	35	0.038	2.27	0.57	0.43	0.50
204	104	0.035	2.93	0.41	0.28	0.47
205	101	0.07	2.27	0.45	0.33	0.39
206	27	0.059	3.63	0.82	0.39	0.94
207	20	0.02	1.58	0.56	0.43	0.43
208	57	0.022	2.73	0.56	0.38	0.54

Table 14-54: Descriptive Statistics for 2m Capped Composites

Domain	Count	Min	Max	Mean	StDev	CV
201	195	0	0.99	0.10	0.04	0.15
202	200	0.001	0.29	0.04	0.03	0.04
203	35	0.003	0.15	0.04	0.03	0.04
204	104	0.007	0.32	0.08	0.06	0.06
205	101	0.001	0.26	0.05	0.04	0.05
206	27	0.001	0.43	0.03	0.00	0.08
207	20	0.001	0.16	0.02	0.01	0.04
208	57	0.001	0.24	0.01	0.00	0.03

Table 14-55: Descriptive Statistics for 2m Capped Composites

Domain	Count	Min	Max	Mean	StDev	CV
201	195	0.1	13.32	1.24	0.48	2.03
202	200	0.2	3.87	0.55	0.33	0.55
203	35	0.25	2.93	0.63	0.40	0.62
204	104	0.17	5.35	1.21	0.95	0.94
205	101	0.25	1.97	0.54	0.37	0.40
206	27	0.2	3.00	0.73	0.30	0.82
207	20	0.2	2.90	0.82	0.42	0.81
208	57	0.2	10.00	1.57	0.50	2.38

Density Assignment

A total of 8,525 density measurements were collected by Troilus from drill core during the 2019 -2020 drill program. Of this total, 1,222 measurements are attributed to the eight domains of the SW Zone and mean densities were assigned to each domain.

Table 14-56 shows the densities assigned to each mineralized domain in the SW Zone. Table 14-57 shows the descriptive statistics for the SW Zone by domain.

Table 14-56: Assigned Densities – SW Zone

Domain	Density (g/cm3)
201	2.81
202	2.72
203	2.72
204	2.91
205	2.80
206	2.84
207	2.75
208	2.79

Table 14-57: Descriptive Statistics for Density by Domain – SW Zone

Domain	Count	Min	Max	Mean	Median	StDev	CV
201	317	2.64	3.85	2.81	2.80	0.10	0.04
202	373	2.42	2.97	2.72	2.70	0.07	0.03
203	69	2.66	2.83	2.72	2.71	0.04	0.01
204	147	2.08	3.07	2.91	2.92	0.11	0.04
205	195	2.69	2.98	2.80	2.80	0.05	0.02
206	7	2.68	2.93	2.84	2.84	0.09	0.03
207	25	2.56	2.81	2.75	2.77	0.06	0.02
208	70	2.66	2.88	2.79	2.80	0.06	0.02

It is the opinion of AGP that the assigned densities are reasonable and acceptable for this resource estimate.

Spatial Analysis

Spatial analysis was performed on 2 m composites on domains 201, 202 and 203. Experimental variograms were established for gold, copper, and silver and were oriented along the overall strike, dip, and across strike directions of the mineralized wireframes.

Table 14-58 to Table 14-60 present the variography parameters for the SW Zone for gold, copper, and silver, respectively.

Table 14-58: Gold Variogram parameters – SW Zone

Sill = 1.00	Search Anisotropy	Az (°)	Dip (°)	Az (°)	X Range (m)	Y Range (m)	Z Range (m)	Variogram Type
C_0 = 0.15								
C_1 = 0.55	Az.Dip.Az.	120	60	0	85	80	2.8	Spherical
C_2 = 0.30	Az.Dip.Az.	120	60	0	120	100	6	Spherical

Table 14-59: Silver Variogram parameters – SW Zone

Sill = 1.00	Search Anisotropy	Az (°)	Dip (°)	Az (°)	X Range (m)	Y Range (m)	Z Range (m)	Variogram Type
C_0 = 0.15								
C_1 = 0.85	Az.Dip.Az.	120	60	0	100	100	12	Spherical

Table 14-60: Silver Variogram parameters – SW Zone

Sill = 1.00	Search Anisotropy	Az (°)	Dip (°)	Az (°)	X Range (m)	Y Range (m)	Z Range (m)	Variogram Type
C_0 = 0.15								
C_1 = 0.85	Az.Dip.Az.	120	60	0	140	110	9	Spherical

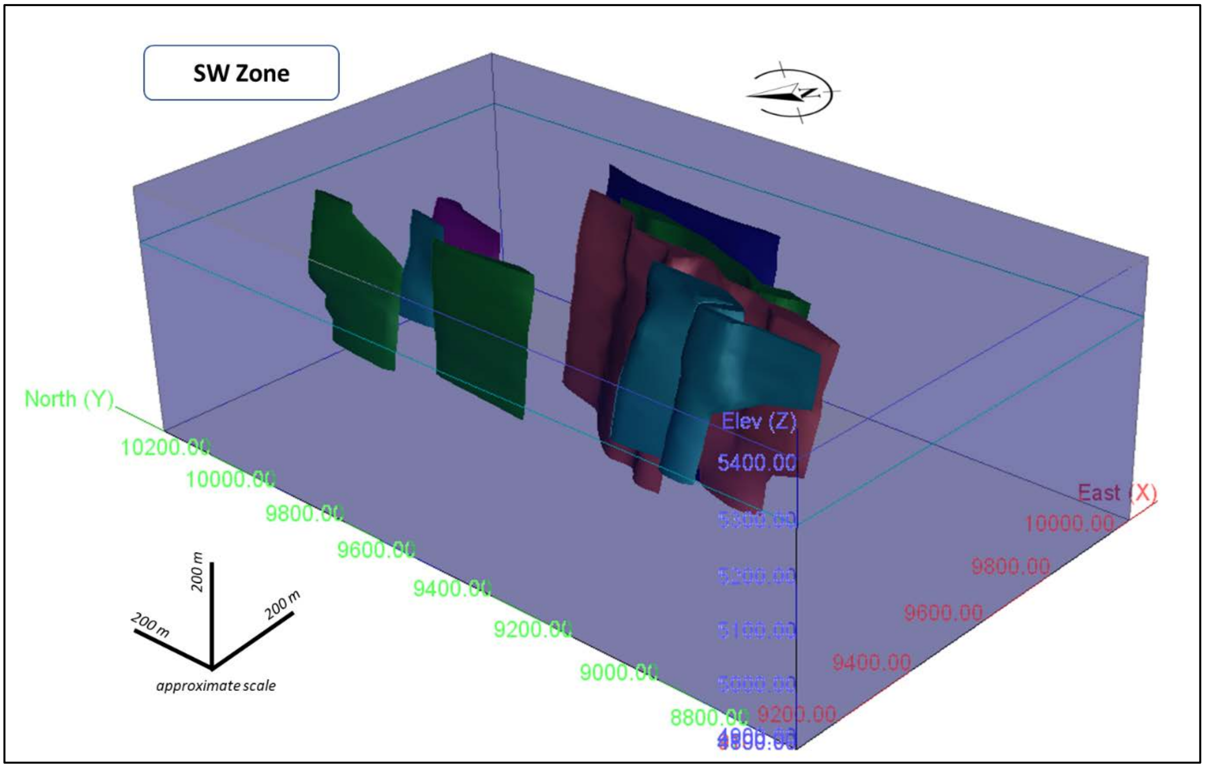
14.6.3 Block Model

The block model for the SW Zone deposit was set up with a block matrix of 10 m long by 10 m long by 10 m high. The block model is not rotated. The block matrix was selected as appropriate for the wide spaced drill pattern and the block height was selected in consideration of an open pit operation. Table 14-61 summarizes the block model parameters and Figure 14-15 presents the block model over the interpreted mineralized domains for the SW Zone.

Table 14-61: Block Model Parameters – SW Zone

	Parameters
Easting	9100 mE
Northing	8700 mN
Maximum Elevation	5420 m
Rotation Angle	No rotation°
Block Size (X, Y, Z in metres)	10 x 10 x 10
Number of blocks in the X direction	95
Number of blocks in the Y direction	165
Number of blocks in the Z direction	53

Figure 14-15: Block Model Parameters – SW Zone



Source : AGP (2020)

The block model extents cover the SW Zone and a minimum of approximately 200 m beyond the interpreted mineralized domains. The block model uses a percent model for each domain.

Block model attributes in the block model includes:

- rock type
- percent (in wireframe)
- density
- metal grades for gold, copper, silver, and calculated gold-equivalent grades for mineralized blocks
- classification
- distance to the nearest composite
- number of composites used in estimation of block
- number of drill holes used for estimation of block
- pass number
- open pit or underground tag

Estimation/Interpolation Methods

The metal grades were interpolated in three passes using the 2 m capped composites. The metal grades were interpolated using OK interpolation method. Variogram parameters for each metal was used in each of these passes and aligned to the domain wireframe. ID2 and NN interpolations were also run for validation purposes.

Each pass required the same minimum and maximum number of composites with a maximum of three composites per drill hole, therefore, two drill holes were required to populate a block. Table 14-62 shows estimation parameters for each pass used to estimate metal grades.

Table 14-62: Estimation Parameters – SW Zone

Pass	Min No Composites	Max No Composites	Max Composites per Drill Holes	Min No of Drill Holes
Pass 1	4	12	3	2
Pass 2	4	12	3	2
Pass 3	4	12	3	2

Each pass increased the search ellipse where Pass 2 was doubled that of Pass 1 and Pass 3 was approximately doubled that of Pass 2. Hard boundaries were kept between all domains and blocks within each domain were estimated only be composites withing the domain wireframe.

Table 14-63 shows search ellipse parameters for each pass used to estimate metal grades.

Table 14-63: Search Ellipse Parameters – SW Zone

Domain	Anisotropy	Azimuth (°)	Dip (°)	Azimuth (°)	Range X (m)	Range Y (m)	Range Z (m)	Search
PASS 1								
201	Az,Dip,Az	110	65	20	30	40	10	Ellipsoidal
202	Az,Dip,Az	110	70	20	30	40	10	Ellipsoidal
203	Az,Dip,Az	110	70	20	30	40	10	Ellipsoidal
204	Az,Dip,Az	100	75	10	30	40	10	Ellipsoidal
205NT	Az,Dip,Az	90	50	10	30	40	10	Ellipsoidal
205N	Az,Dip,Az	90	68	5	30	40	10	Ellipsoidal
205S	Az,Dip,Az	90	70	0	30	40	10	Ellipsoidal
206	Az,Dip,Az	85	57	0	30	40	10	Ellipsoidal
207	Az,Dip,Az	90	75	0	30	40	10	Ellipsoidal
208	Az,Dip,Az	90	80	5	30	40	10	Ellipsoidal
PASS 2	Az,Dip,Az	same as Pass 1			60	80	10	Ellipsoidal
PASS 3	Az,Dip,Az	same as Pass 1			120	145	25	Ellipsoidal

Az,Dip,Az – Azimuth, Dip, Azimtuh

Domain 205 was split into three block selections representing a south, north, and north tip of the wireframe. This was done to honour the changes in orientation of the domain. Current interpretation shows a reverse fold in the wireframe based on one drill hole. Due to the block size it was determined that an unfolding of the wireframe for estimation purposes would not significantly impact the estimation and that further drilling and information is required to confirm the current interpretation or whether this may be a set of offset splays in the domain.

Block Model Validation

The block model was validated using the following methods:

- visual inspection and comparison of block grades with composite and assay grades
- statistical comparison of resource assay and block grade distributions
- inspection of swath plots with composites and block grades elevations and northings

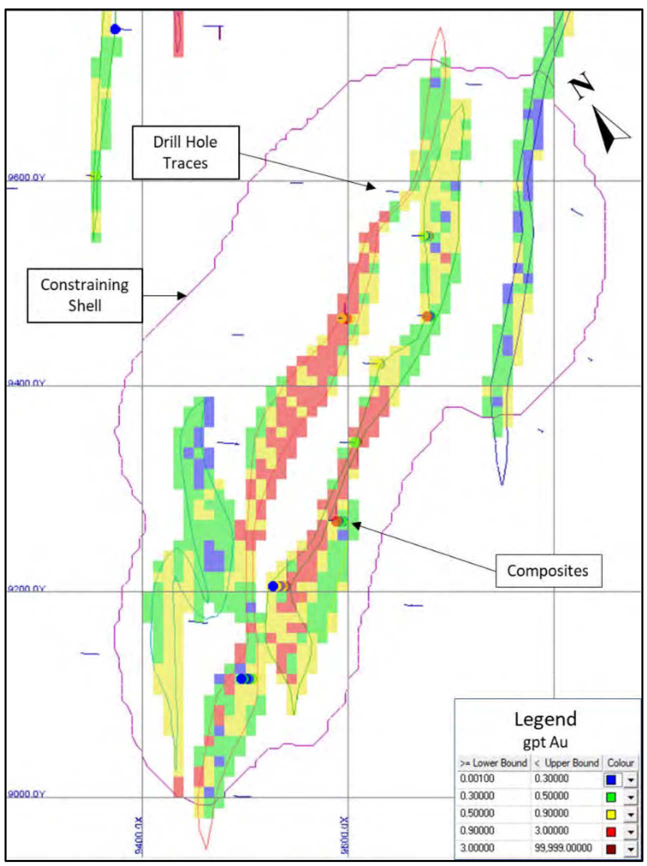
Table 14-64 presents a comparison of the mean gold grades comparing composites to OK, ID2 and NN interpolated mean grades by mineralized domain. AGP is satisfied that the block model gold grades reflect the gold grades from the drill core samples.

Table 14-64: Mean Gold Grades – SW Zone (no zeroes)

	OK	ID	NN	Comp
201	0.83	0.83	0.84	0.91
202	0.63	0.61	0.66	0.65
203	0.49	0.50	0.58	0.64
204	0.42	0.47	0.41	0.50
205	0.50	0.46	0.48	0.46
206	0.94	0.82	0.91	0.82
207	0.42	0.45	0.52	0.56
208	0.59	0.58	0.66	0.56

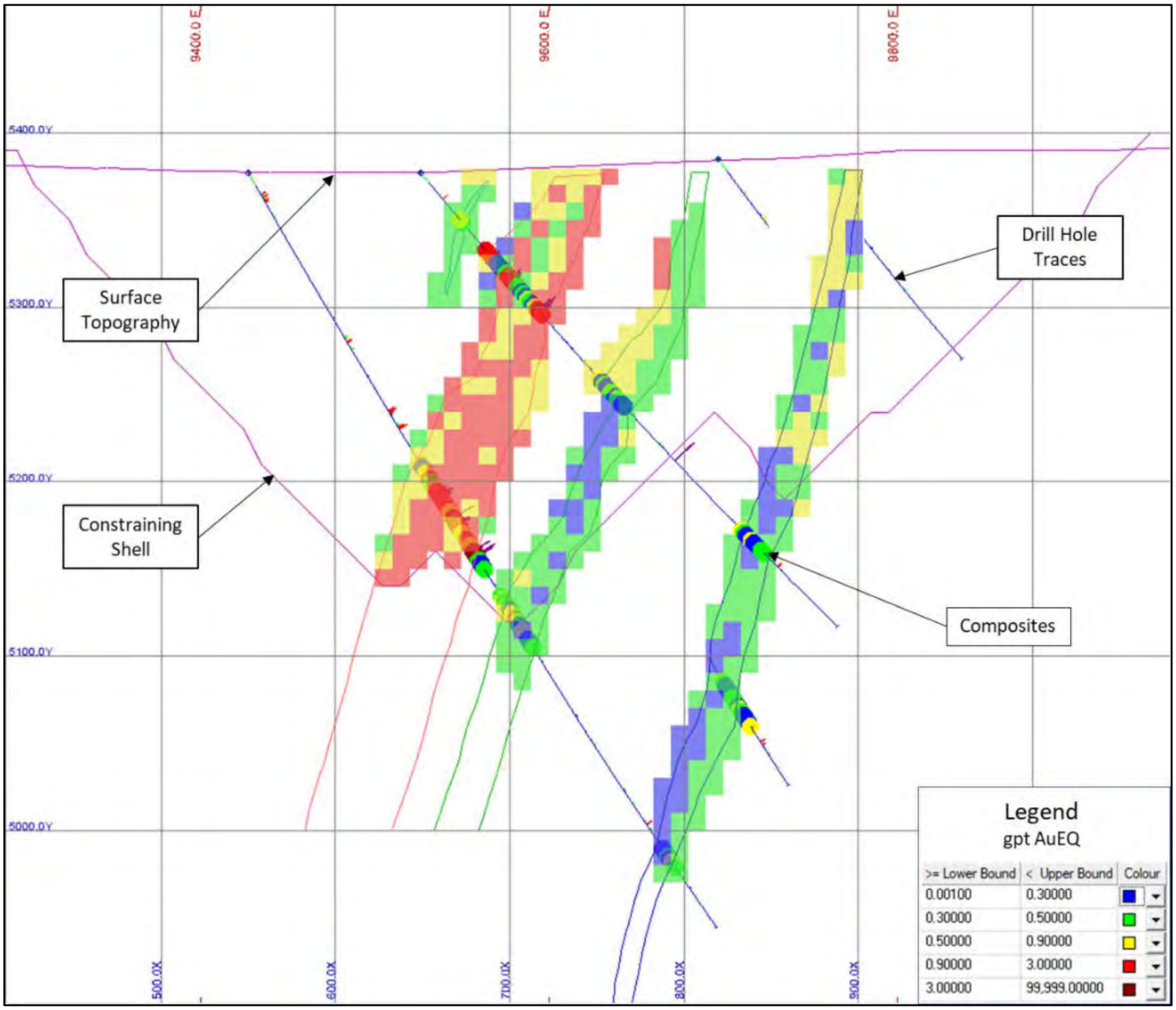
Figure 14-16 and Figure 14-17 present plan section and cross section views for SW Zone (plan view elevation 5,265 and cross section 13,700N).

Figure 14-16: SW Zone - Plan View Elevation 5,205 m



Source : AGP (2020)

Figure 14-17: Z87 Zone - Plan View Elevation 5,005 m



Source : AGP (2020)

Swath plots were reviewed by northing easting and elevation. The distribution of gold and copper composite and interpolated block grades were compared to the OK, ID2 and NN grades. No issues were found with the distribution of interpolated grades. Figure 14-18 to Figure 14-20 present the swath plots for gold in the SW Zone by northing, easting and elevation, respectively.

Figure 14-18: SW Zone – Swath Plot for Gold, by Easting

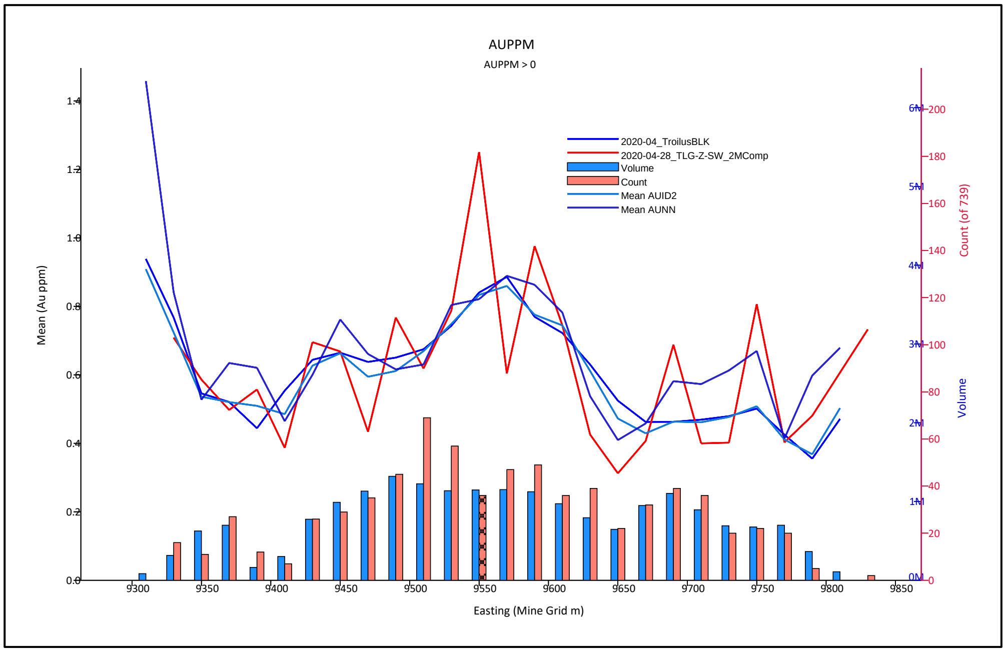


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Figure 14-19: SW Zone – Swath Plot for Gold, by Easting

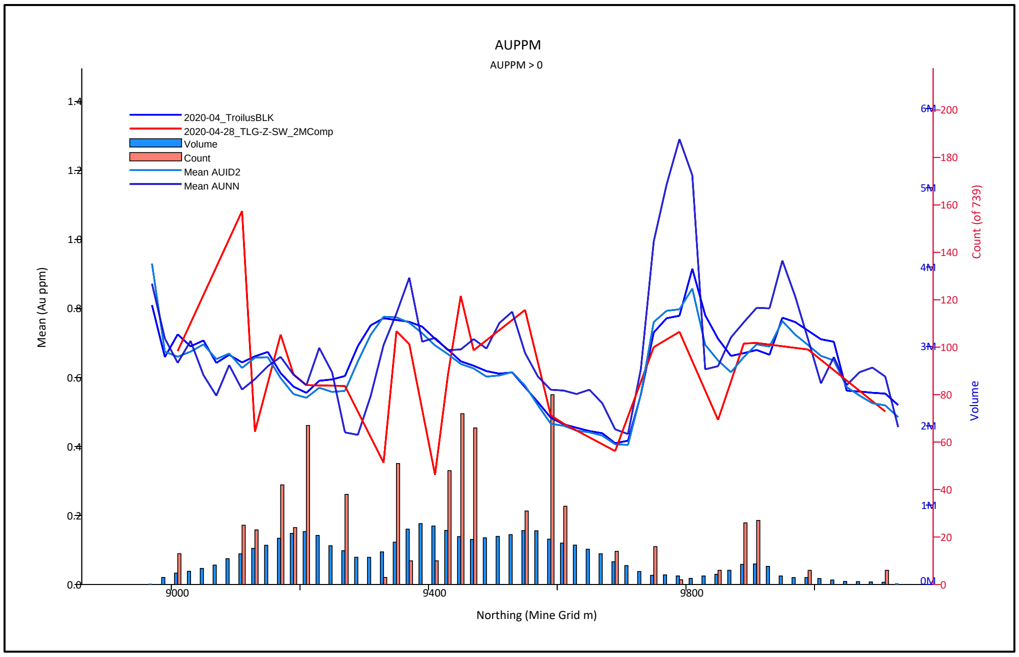
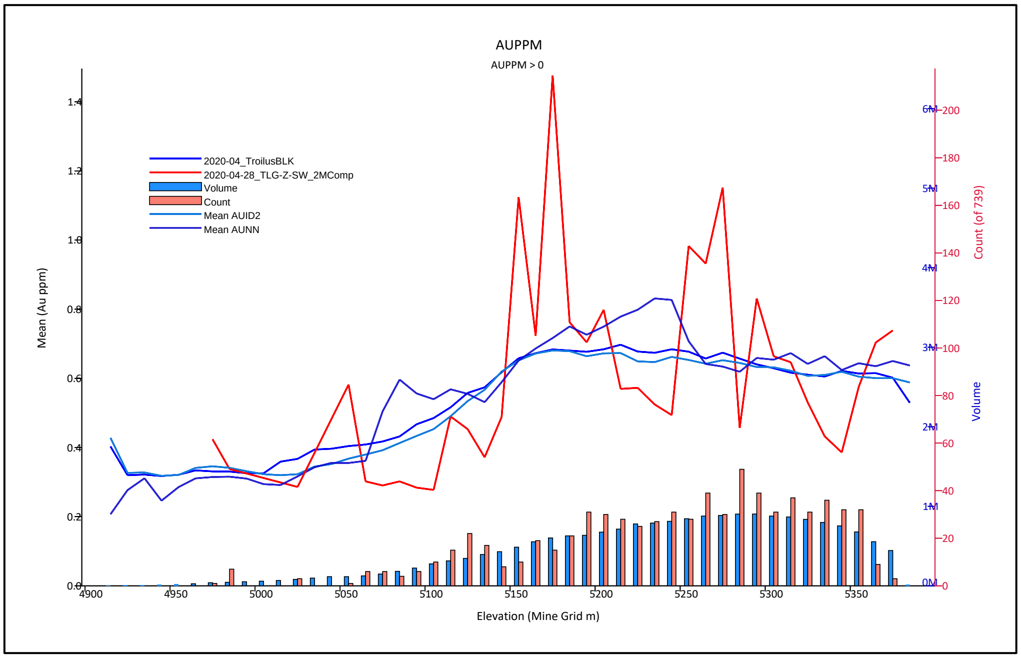


Figure 14-20: SW Zone – Swath Plot for Gold, by Easting



14.7 Mineral Resources

14.7.1 Classification of Mineral Resources

Classification

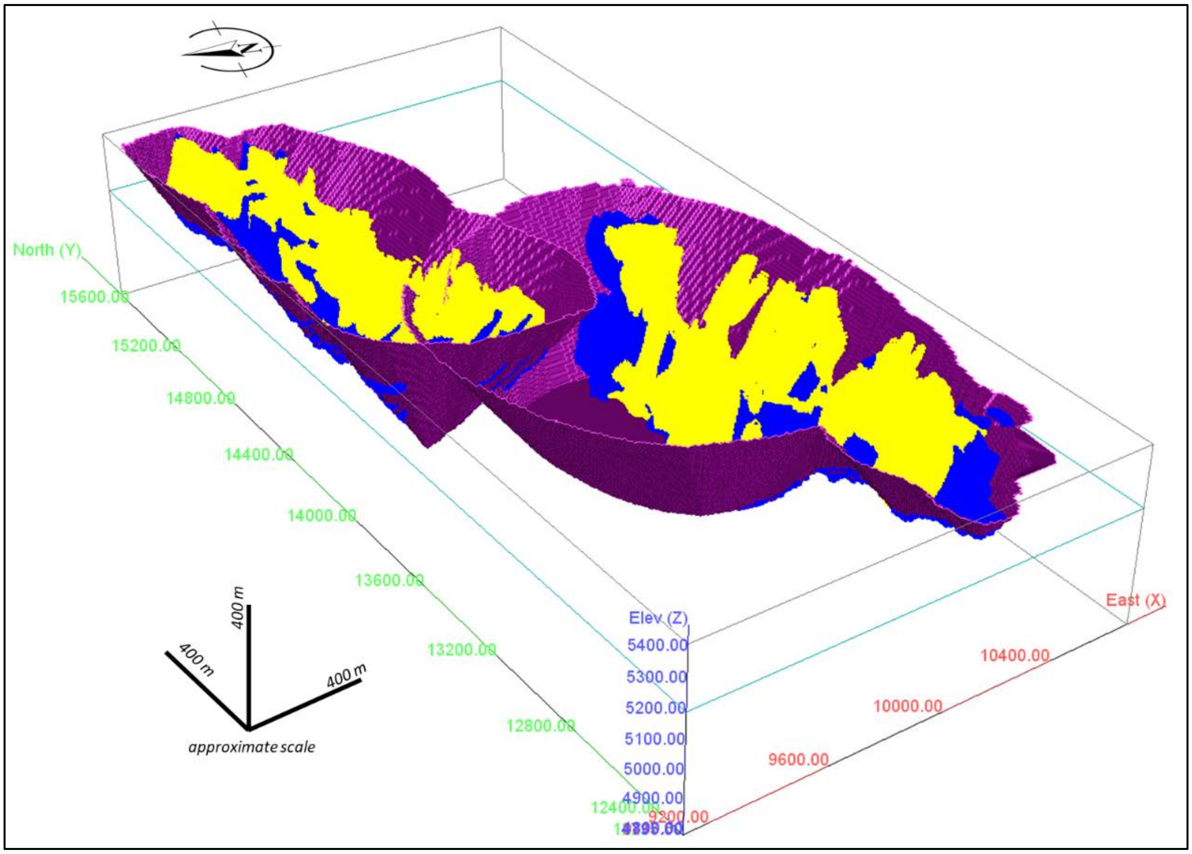
Definitions for Mineral Resource categories used in this report are consistent with those defined by CIM (2014) and referenced by NI 43-101. In the CIM classification, a Mineral Resource is defined as "a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction". Mineral Resources are classified into Measured, Indicated, and Inferred categories. A Mineral Reserve is defined as the "economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study". Mineral Reserves are classified into Proven and Probable categories. No Mineral Reserves have been estimated for Project.

Z87 and J4/J5 Zones

For the Z87 and J4/J5 Zones, blocks interpolated in the first pass, requiring at least two holes, and within 60 m from a drill hole were initially considered for classification into the Indicated Mineral Resource category. A manual contour was then digitized, on a domain by domain basis, consolidating the areas with contiguous candidate blocks and discarding isolated blocks or patches of blocks retained with the numerical approach. The manual contours were used to classify the blocks retained inside the contours into the Indicated Mineral Resource category. Out of the remaining interpolated blocks, those within 120 m from a drill hole were classified into the Inferred Mineral Resource category.

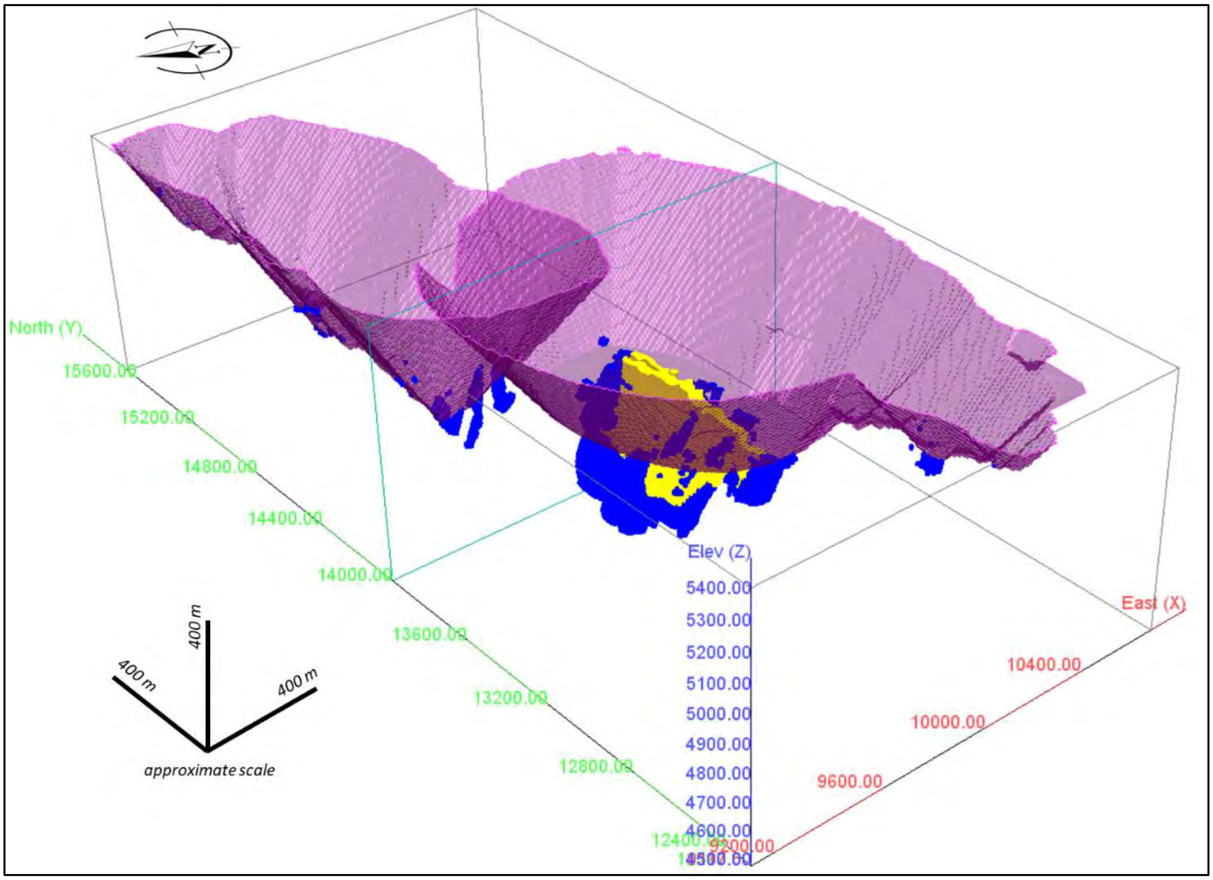
Figure 14-21 and Figure 14-22 present the open pit resource classified blocks in Z87 Zone and J4/J5 Zone, respectively. the underground classified blocks in Z87 Zone. Figure 14-23 presents the open classified blocks in the SW Zone.

Figure 14-21: Open Pit Mineral Resources – Z87 Zone and J4/J5



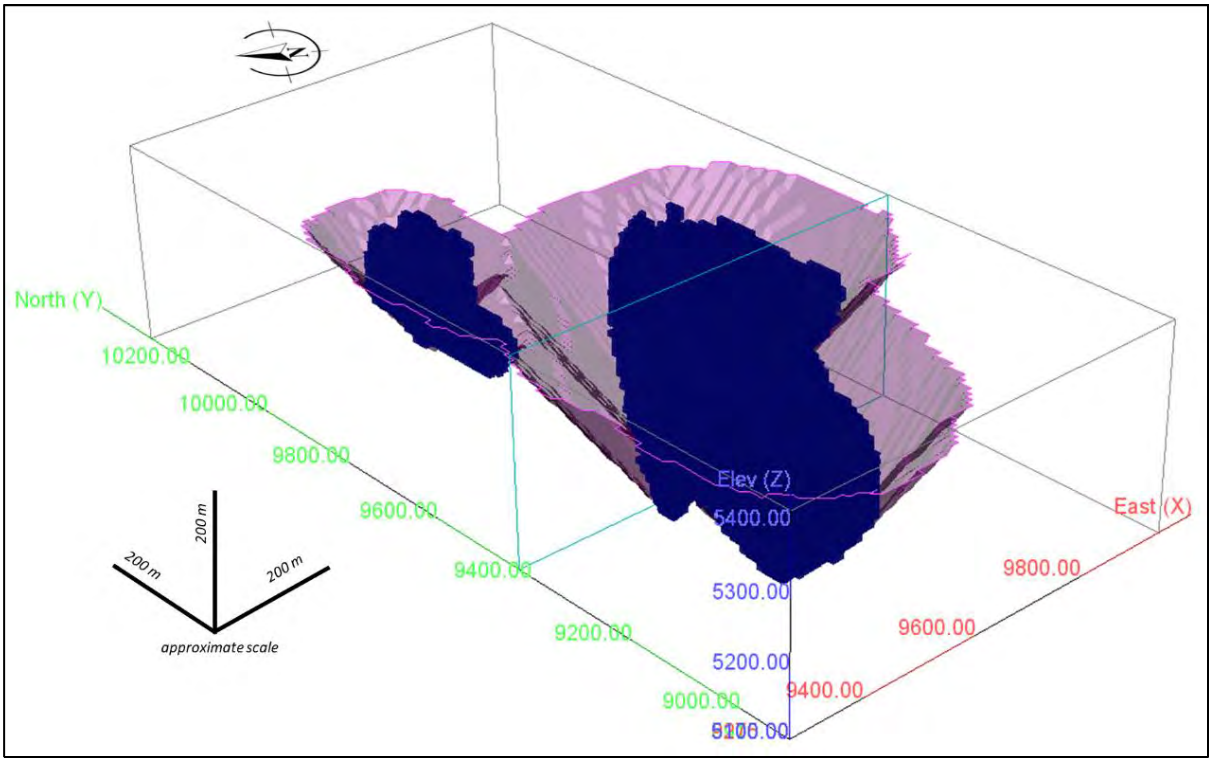
yellow – Indicated; blue - Inferred
Source: AGP 2020

Figure 14-22: Underground Mineral Resources – Z87 Zone and J4/J5 Zone



yellow – Indicated; blue - Inferred
Source: AGP 2020

Figure 14-23: Open Pit Mineral Resources – SW Zone



blue - Inferred
Source: AGP 2020

14.7.2 Reasonable Prospects of Economic Extraction

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.

For the Z87 and J4/J5 Zones, blocks interpolated in the first pass with a minimum of two drill holes, and a nearest distance of 60 m were initially classified as Indicated resources. Polylines were made, on a lens by lens basis, to consolidate contiguous blocks and downgrading isolated blocks. The manual contours were used to classify the blocks retained inside the contours into the Indicated Mineral Resource category. Block interpolated with a nearest distance of 120 m from a drill hole were classified as Inferred resources.

For the SW Zone, Blocks were classified as Inferred resources estimated with a minimum of two drill holes and with a nominal distance to the closest point of less than 120 m. Very few blocks were populated on the first pass, therefore, these blocks were included as Inferred Resources.

Metal Equivalent

A metal equivalent grade was used to determine cut-off grades for the Troilus 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 Troilus 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 Zones. The AuEQ grades were calculated for each block after metal grade interpolations were completed using the following:

$$Z87\ Zone \quad AuEq = Au\ grade + (1.2566\ x\ Cu\ grade) + (0.0103\ x\ Ag\ grade)$$

$$J4/J5\ Zone \quad AuEq = Au\ grade + (1.2979\ x\ Cu\ grade) + (0.0108\ x\ Ag\ grade)$$

$$SW\ Zone \quad AuEq = Au\ grade + (1.2768\ x\ Cu\ grade) + (0.0106\ x\ Ag\ grade)$$

The parameters used in the above formula are listed in Table 14-65.

Table 14-65: Parameters for the AuEQ Formula

Metal	Price ($US) All Zones	Recovery (%) Z87	J4/J5	SW
Gold	$1,600.00/oz.	92	88	90
Copper	$3.25/lb	83	82	82.5
Silver	$20.00/oz.	76	76	76

The metal prices used are based on consensus, the three-year rolling average between 12 May 2017 and 12 May 2020 and metal forecasts. The metal recoveries is based on historic recoveries at the Troilus Mine.

Cut-off Grade

For all Zones at the Troilus Project, AGP has determined a resource cut-off grade of 0.3 gpt AuEQ to be used for reporting of the mineral resources within constraining shells for the material amenable to open pit extraction. For the Z87 Zone and J4/J5 Zone, a resource cut-off grade of 0.9 gpt AuEQ for material that may be amenable to underground extraction, for contiguous blocks below the constraining shells. The cut-off grades are based on the parameters defined below.

Constraining Shell Parameters

The block models were exported for further economic analysis by Willie Hamilton, Open Pit Mine Engineer with AGP. The model was exported in ASCII format and imported into Hexagon MineSight® for use in developing the constraining shell for the reported mineral resources. Table 14-66 shows the economic assumptions made to constrain the reported mineral resources at the Project.

Table 14-66: Parameters for Constraining Shells, by Zone

Parameter	Units	Z87	J4/J5	SW (HG/LG)
Metal Prices				
Au	$US/oz Au	1600.00	1600.00	1600.00
Cu	$US/lb Cu	3.25	3.25	3.25
Ag	$US/oz Ag	20.00	20.00	20.00
Metal Recoveries				
Au	%	88/90	88/90	88/90
Cu	%	90/92	90/92	90/92
Ag	%	40	40	40
$CAD: $US		1.30	1.30	1.30
Mining Rate – OP	tpd	35,000	35,000	35,000
Mining Cost – OP	$US/t total	1.71	1.71	1.66
Mining Cost – UG	$US/t total			-
Processing Cost	$/t mill feed	6.99	6.99	6.99
G&A Cost	$/t mill feed	1.45	1.45	1.45
OP Wall Slope Angles				
030°Az	degrees	56	60	56
0450°Az	degrees	49.5	50	49.5
135°Az	degrees	49.5	50	49.5
150°Az	degrees	56	60	56
210°Az	degrees	56		56
225°Az	degrees	53		53
315°Az	degrees	53		53
330°Az	degrees	56		56

OP – Open Pit; UG – Underground; G&A – General and Administration
HG – high grade Au >= 1.2 gpt Au; Cu > 0.13 %Cu
LG – low grade Au < 1.2 gpt Au; Cu <= 0.13 %Cu

14.8 Mineral Resource Statement

14.8.1 Mineral Resource Statement

The mineral resources for the Troilus Project are: Indicated Resources of 177.1 Mt at 0.75 gpt Au, 0.08 %Cu, 1.17 gpt Ag and 0.87 gpt AuEQ; and Inferred Resources of 116.7 Mt at 0.32 gpt Au, 0.05 %Cu, 0.69 gpt Ag and 0.36 gpt AuEQ. The effective date of the Mineral Resources is 20 July 2020.

Table 14-67 presents the Mineral Resources for the combined mineral resources amenable to open pit and underground resources for the Troilus Project.

Table 14-67: Mineral Resources for the Troilus Project; combined open pit and underground resources

Classification	Tonnes (,000t)	Grade				Contained Metal			
		Au (gpt Au)	Cu (% Cu)	Ag (gpt Ag)	AuEQ (gpt AuEQ)	Au (Moz)	Cu (Mlbs)	Ag (Moz)	AuEQ (M oz)
Indicated	177.3	0.75	0.08	1.17	0.87	4.30	322.60	6.66	4.96
Inferred	116.7	0.73	0.07	1.04	0.84	2.76	189.73	3.91	3.15

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 gpt AuEQ where the metal equivalents were calculated as follows:
    - Z87 Zone AuEq = Au grade + 1.2566 * Cu grade + 0.0103 * Ag grade
    - J4/J5 Zone AuEq = Au grade + 1.2979 * Cu grade + 0.0108 * Ag grade
    - SW Zone AuEq = Au grade + 1.2768 * Cu grade + 0.0106 * Ag grade
- Metal prices for the AuEQ formulas are: $US 1,600/ oz Au; $3.25/lb Cu, and $20.00/ oz Ag; with an exchange rate of US$1.00: CAD$1.30
- Metal recoveries for the AuEQ formulas are:
    - Z87 Zone 83% for Au recovery, 92% for Cu recovery and 76% for Ag recovery
    - J4J5 Zone 82% for Au recovery, 88% for Cu recovery and 76% for Ag recovery
    - Z87 Zone 82.5% for Au recovery, 90% for Cu recovery and 76% for Ag recovery
- Underground cut-off grade is 0.9 AuEQ at Z87 Zone below constraining pit
- Capping of grades varied between 2.00 g/t Au and 26.00 g/t Au; between 1.00 g/t Ag and 20.00 g/t Ag on raw assays; and 1.00 %Cu on raw assays
- The density varies between 2.72 g/cm^3 and 2.91 g/cm^3 depending on mineralized zone
- Capping of grades varied between 2.00 g/t Au and 26.00 g/t Au; between 1.00 g/t Ag and 20.00 g/t Ag on raw assays; and 1.00 %Cu on raw assays in SW Zone
- The density varies between 2.72 g/cm^3 and 2.91 g/cm^3 depending on mineralized zone

14.8.2 Open Pit Mineral Resources

The mineral resources for the Troilus Project deposit amenable to open pit extraction at a 0.3 gpt AuEQ cut-off grade are: an Indicated Resource of 164.2 Mt at 0.68 g/t Au, 0.08 %Cu, 1.20 gpt Ag and 0.80 gpt AuEQ; and an Inferred Resource of 101.2 Mt at 0.60 g/t Au, 0.07 %Cu, 1.12 gpt Ag and 0.70 gpt AuEQ.

Table 14-68 presents the Mineral Resources amenable to open pit extraction

Table 14-68: Open Pit Mineral Resources for Troilus Project at a 0.3 gpt AuEQ Cut-off Grade – All Deposits

Classification	Tonnes (,000t)	Grade				Contained Metal			
		Au (gpt Au)	Cu (% Cu)	Ag (gpt Ag)	AuEQ (gpt AuEQ)	Au (Moz)	Cu (Mlbs)	Ag (Moz)	AuEQ (M oz)
Z87 Zone									
Indicated	84.6	0.79	0.09	1.39	0.92	2.15	169.54	3.77	2.50
Inferred	32.7	0.60	0.07	1.50	0.70	0.63	49.34	1.57	0.73
J4/J5 Zone									
Indicated	79.6	0.57	0.07	1.00	0.67	1.47	115.16	2.55	1.71
Inferred	45.9	0.55	0.07	0.96	0.65	0.82	65.94	1.42	0.96
SW Zone									
Inferred	22.6	0.70	0.07	0.89	0.80	0.51	35.73	0.65	0.58
TOTALS									
Indicated	**164.2**	**0.68**	**0.08**	**1.20**	**0.80**	**3.62**	**284.69**	**6.32**	**4.21**
Inferred	**101.2**	**0.60**	**0.07**	**1.12**	**0.70**	**1.95**	**151.01**	**3.65**	**2.27**

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 gpt AuEQ
- AuEQ equivalents were calculated as follows:
  - Z87 Zone AuEq = Au grade + 1.2566 * Cu grade + 0.0103 * Ag grade
  - J4/J5 Zone AuEq = Au grade + 1.2979 * Cu grade + 0.0108 * Ag grade
  - SW Zone AuEq = Au grade + 1.2768 * Cu grade + 0.0106 * Ag grade
- Metal prices for the AuEQ formulas are: $US 1,600/ oz Au; $3.25/lb Cu, and $20.00/ oz Ag; with an exchange rate of US$1.00: CAD$1.30
- Metal recoveries for the AuEQ formulas are:
  - Z87 Zone 83% for Au recovery, 92% for Cu recovery and 76% for Ag recovery
  - J4J5 Zone 82% for Au recovery, 88% for Cu recovery and 76% for Ag recovery
  - SW Zone 82.5% for Au recovery, 90% for Cu recovery and 76% for Ag recovery
- Capping of grades varied between 2.00 g/t Au and 26.00 g/t Au; between 1.00 g/t Ag and 20.00 g/t Ag on raw assays; and 1.00 %Cu on raw assays
- The density varies between 2.72 g/cm^3 and 2.91 g/cm^3 depending on mineralized zone or domain

14.8.3 Underground Mineral Resources

The mineral resources for the Troilus Project deposit amenable to underground extraction are: An Indicated Resource of 13.1 Mt at 1.61 g/t Au, 0.13 %Cu, 0.81 gpt Ag and 1.79 gpt AuEQ; and an Inferred Resource of 15.5 Mt at 1.62 g/t Au, 0.1 %Cu, 0.52 gpt Ag and 1.77 gpt AuEQ.

Table 14-69 presents the Mineral Resources amenable to underground extraction.

Table 14-69: Underground Mineral Resources for the Troilus Project at a 0.9 gpt AuEQ Cut-off Grade – Z87 Zone

Classification	Tonnes (,000t)	Grade				Contained Metal			
		Au (gpt Au)	Cu (% Cu)	Ag (gpt Ag)	AuEQ (gpt AuEQ)	Au (Moz)	Cu (Mlbs)	Ag (Moz)	AuEQ (M oz)
Z87 Zone									
Indicated	13.1	1.61	0.13	0.81	1.79	0.68	37.90	0.34	0.75
Inferred	13.5	1.70	0.12	0.37	1.85	0.74	34.48	0.16	0.80
J4/J5 Zone									
Indicated	0.01	1.03	0.03	0.47	1.07	0.0002	0.01	0.0001	0.0003
Inferred	2.0	1.06	0.10	1.55	1.21	0.07	4.24	0.10	0.08
TOTALS									
Indicated	**13.1**	**1.61**	**0.13**	**0.81**	**1.79**	**0.68**	**37.91**	**0.34**	**0.75**
Inferred	**15.5**	**1.62**	**0.11**	**0.52**	**1.77**	**0.81**	**38.72**	**0.26**	**0.88**

Notes:

- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
- Summation errors may occur due to rounding
- Underground cut-off grade is 0.9 gpt AuEQ
- AuEQ equivalents were calculated as follows:
  - Z87 Zone AuEq = Au grade + 1.2566 * Cu grade + 0.0103 * Ag grade
  - J4/J5 Zone AuEq = Au grade + 1.2979 * Cu grade + 0.01083 * Ag grade
- Metal prices for the AuEQ formulas are: $US 1,600/ oz Au; $3.25/lb Cu, and $20.00/ oz Ag; with an exchange rate of US$1.00: CAD$1.30
- Metal recoveries for the AuEQ formulas are:
  - Z87 Zone 83% for Au recovery, 92% for Cu recovery and 76% for Ag recovery
  - J4/J5 Zone 82% for Au recovery, 88% for Cu recovery and 76% for Ag recovery
- Capping of grades varied between 5.00 g/t Au and 26.00 g/t Au: between 10.00 g/t Ag and 20.00 g/t Ag on raw assays
- The density of the mineralized domains at Z87 Zone is 2.86 g/cm^3: and 2.77 and 2.78 g/cm^3 at J4/J5 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 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.

14.8.4 Grade Sensitivity

The Mineral Resources for the Project are reported below to demonstrate the sensitivity to various AuEQ cut-off grades for each Zone.

Z87 Zone

Table 14-70 and Table 14-71 present the deposit sensitivity to various AuEQ cut-off grades in Z87 for the open pit and underground Mineral Resources, respectively.

Table 14-70: Open Pit Indicated and Inferred Mineral Resources for the Z87 Zone; at various cut-off grades

Cut-off (gpt AuEQ)	Tonnage ('000 t)	Au (gpt)	Cu (%)	Ag (gpt)	AuEQ (gpt)	Contained Au (oz Au)	Contained Cu (lb Cu)	Contained Ag (oz Au)	Contained AuEQ (oz Au)
Indicated									
0.6	48.4	1.10	0.12	1.62	1.26	1,709,000	127,092,000	2,527,000	1,968,000
0.5	60.7	0.97	0.11	1.53	1.12	1,890,000	143,283,000	2,980,000	2,183,000
0.4	73.8	0.86	0.10	1.45	1.00	2,047,000	158,973,000	3,439,000	2,374,000
0.3	84.6	0.79	0.09	1.39	0.92	2,147,000	169,537,000	3,769,000	2,497,000
0.2	88.3	0.76	0.09	1.36	0.89	2,172,000	172,433,000	3,868,000	2,528,000
0.1	88.8	0.76	0.09	1.36	0.89	2,174,000	172,663,000	3,878,000	2,530,000
Inferred									
0.6	15.3	0.86	0.08	1.64	0.98	421,000	27,072,000	805,000	479,000
0.5	21.1	0.74	0.08	1.59	0.86	506,000	35,506,000	1,076,000	582,000
0.4	27.5	0.66	0.07	1.52	0.76	580,000	43,626,000	1,342,000	674,000
0.3	32.7	0.60	0.07	1.50	0.70	627,000	49,340,000	1,574,000	733,000
0.2	34.3	0.58	0.07	1.47	0.68	637,000	50,669,000	1,619,000	746,000
0.1	34.6	0.57	0.07	1.46	0.67	638,000	50,876,000	1,625,000	748,000

Table 14-71: Underground Indicated and Inferred Mineral Resources for the Z87 Zone; at various cut-off grades

Cut-off (gpt AuEQ)	Tonnage ('000 t)	Au (gpt)	Cu (%)	Ag (gpt)	AuEQ (gpt)	Contained Au (oz Au)	Contained Cu (lb Cu)	Contained Ag (oz Au)	Contained AuEQ (oz Au)
Indicated									
1.1	11.0	1.75	0.14	0.88	1.93	619,000	33,663,000	313,000	684,000
1.0	12.2	1.67	0.13	0.83	1.85	656,000	36,304,000	328,000	726,000
0.9	13.1	1.61	0.13	0.81	1.79	680,000	37,904,000	340,000	753,000
0.8	13.7	1.58	0.13	0.79	1.75	694,000	38,827,000	348,000	768,000
0.7	14.0	1.55	0.13	0.79	1.72	701,000	39,350,000	355,000	777,000
Inferred									
1.1	9.3	2.09	0.12	0.45	2.25	622,000	24,857,000	134,000	669,000
1.0	10.8	1.92	0.12	0.42	2.07	668,000	28,403,000	148,000	722,000
0.9	13.5	1.70	0.12	0.37	1.85	740,000	34,479,000	162,000	804,000
0.8	14.0	1.66	0.12	0.37	1.81	751,000	35,587,000	165,000	818,000
0.7	14.2	1.65	0.11	0.36	1.80	754,000	35,850,000	166,000	821,000

J4/J5 Zone

Table 14-72 and Table 14-73 present the deposit sensitivity to various AuEQ cut-off grades in the J4/J5 Zone for the open pit and underground Mineral Resources, respectively.

Table 14-72: Open Pit Indicated and Inferred Mineral Resources for the J4/J5 Zone; at various cut-off grades

Cut-off (gpt AuEQ)	Tonnage ('000 t)	Au (gpt)	Cu (%)	Ag (gpt)	AuEQ (gpt)	Contained Au (oz Au)	Contained Cu (lb Cu)	Contained Ag (oz Au)	Contained AuEQ (oz Au)
Indicated									
0.6	34.9	0.85	0.07	1.15	0.95	953,000	54,892,000	1,289,000	1,071,000
0.5	48.3	0.74	0.07	1.11	0.84	1,150,000	73,729,000	1,721,000	1,308,000
0.4	65.0	0.64	0.07	1.05	0.74	1,341,000	96,515,000	2,198,000	1,547,000
0.3	79.6	0.57	0.07	1.00	0.67	1,469,000	115,157,000	2,554,000	1,714,000
0.2	84.9	0.55	0.06	0.97	0.64	1,502,000	120,759,000	2,655,000	1,759,000
0.1	85.7	0.55	0.06	0.97	0.64	1,505,000	121,259,000	2,666,000	1,763,000
Inferred									
0.6	19.6	0.82	0.07	1.11	0.92	516,000	29,271,000	699,000	579,000
0.5	27.2	0.72	0.07	1.07	0.81	626,000	40,545,000	939,000	713,000
0.4	37.4	0.62	0.07	1.01	0.71	743,000	54,888,000	1,221,000	860,000
0.3	45.9	0.55	0.07	0.96	0.65	816,000	65,939,000	1,423,000	956,000
0.2	49.2	0.53	0.06	0.94	0.62	836,000	69,954,000	1,488,000	985,000
0.1	49.7	0.52	0.06	0.94	0.62	838,000	70,194,000	1,494,000	987,000

Table 14-73: Underground Indicated and Inferred Mineral Resources for the J4/J5 Zone; at various cut-off grades

Cut-off (gpt AuEQ)	Tonnage ('000 t)	Au (gpt)	Cu (%)	Ag (gpt)	AuEQ (gpt)	Contained Au (oz Au)	Contained Cu (lb Cu)	Contained Ag (oz Au)	Contained AuEQ (oz Au)
Indicated									
1.1	0.002	1.26	0.06	0.52	1.35	80	3,000	30	90
1.0	0.002	1.26	0.06	0.52	1.35	80	3,000	30	90
0.9	0.007	1.03	0.03	0.47	1.07	240	5,000	110	250
0.8	0.010	0.96	0.03	0.56	1.01	310	8,000	180	330
0.7	0.029	0.77	0.04	0.64	0.83	720	25,000	600	770
Inferred									
1.1	1.1	1.23	0.10	1.57	1.38	43,000	2,452,000	55,000	48,000
1.0	1.5	1.15	0.10	1.56	1.30	54,000	3,244,000	73,000	61,000
0.9	2.0	1.06	0.10	1.55	1.21	67,000	4,237,000	97,000	76,000
0.8	2.6	0.99	0.09	1.43	1.12	82,000	5,128,000	119,000	93,000
0.7	3.6	0.89	0.08	1.34	1.01	103,000	6,566,000	156,000	118,000

SW Zone

Table 14-74 present the deposit sensitivity to various AuEQ cut-off grades in Z87 for the open pit and underground Mineral Resources, respectively.

Table 14-74: Open Pit Inferred Mineral Resources for the Z87 Zone; at various cut-off grades

Cut-off (gpt AuEQ)	Tonnage ('000 t)	Au (gpt)	Cu (%)	Ag (gpt)	AuEQ (gpt)	Contained Au (oz Au)	Contained Cu (lb Cu)	Contained Ag (oz Au)	Contained AuEQ (oz Au)
Inferred									
0.6	12.0	0.97	0.09	1.12	1.10	374,000	25,082,000	435,000	425,000
0.5	15.7	0.85	0.08	1.02	0.97	430,000	29,139,000	516,000	490,000
0.4	20.1	0.75	0.08	0.94	0.86	485,000	33,596,000	604,000	554,000
0.3	22.6	0.70	0.07	0.89	0.80	509,000	35,731,000	649,000	583,000
0.2	23.3	0.69	0.07	0.88	0.79	514,000	36,170,000	659,000	589,000
0.1	23.3	0.69	0.07	0.88	0.79	514,000	36,172,000	659,000	589,000

14.9 Factors That May Affect the Mineral Resource Estimate

In order to constrain the mineralization, current geological domains capture the mineralization within a 0.3 gpt AuEQ limit. However, it has been observed in plan views that several of the domains do have a 'saw-tooth' shape in some areas in both Z87 and J4/J5 Zones. It was recommended that the mineralized domains be revisited in both plan and cross-sections to better reflect the overall trend of the mineralization in these areas. At the time of writing, this is being carried out. This is anticipated to have a relatively small impact on the mineral resources and will show a more accurate representation of the mineralization.

In the SW Zone, domain 205 has been interpreted as a reverse fold based on the intersection in one drill hole. While a reverse fold may exist, further drilling will be able to confirm this interpretation. Additional drilling in this zone is recommended and is expected to provide a higher degree of confidence in the continuity of geology and mineral resources are anticipated to expand, but not decrease, in this zone.

15 MINERAL RESERVE ESTIMATES

The Troilus Project is at a PEA level of study and therefore currently has no reserves.

16 MINING METHODS

16.1 Introduction

The Troilus Project (Project) is located in central Quebec and is situated approximately 120 km north of Chibougamau. The Z87 Zone and 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 three principal mineralized zones: Z87, J4/J5 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, with a transition to underground mining below the Z87 upper zone. This is based on the size of the resource, tenor of the grade, grade distribution and proximity to topography for the deposits.

The combined mine schedule for open pit and underground mining consists of 192.5 Mt of mill feed grading 0.71 g/t gold, 0.076% copper, and 0.97 g/t silver over a mine life of twenty-two years. Open pit waste tonnage totals 591 Mt and will be placed into waste storage areas. The overall open pit strip ratio is 3.94:1. The mine schedule utilizes open pit and underground mining areas to supply mill feed up to a maximum of 12.6 Mtpa to the mill facility.

The current mine life includes one year of pre-stripping followed by twenty-two years of mining. Mill feed is stockpiled during the pre-production year and reaches a peak stockpile capacity of 8.9 Mt near the end of year 6.

The open pit mining starts in Year -1 and continues uninterrupted until Year 14. Underground development is initiated in Year 6 with underground mill feed becoming available in Year 8 and continuing until Year 22.

16.2 Mining Geotechnical

AGP has completed a compilation, review, and preliminary mining-geotechnical assessment of the Troilus property's geology, geotechnical, and hydro-geological data to support the proposed PEA-level open pit and underground mine designs. This sub-section (Section 16.2) contains a brief summary of the key data, a description of the project geotechnical model, as confirmed by AGP, and preliminary open pit and underground excavation design criteria and ground control guidance, for ongoing mine planning and cost estimating tasks.

A gap analysis has also been completed by AGP to provide recommendations for further field studies (geotechnical drilling, logging, in-hole testing, field-index testing, laboratory strength testing), and geotechnical and hydro-geological design work, necessary to advance the project to the PFS-level. Mining-geotechnical uncertainties and opportunities related to the project are also identified and described, below.

16.2.1 Site Inspection

AGP's J. Roland Tosney completed a 'Qualified Person' (QP) site inspection of the Troilus Property from December 11th – 14th, 2019.

Tasks completed during Mr. Tosney's site inspection included:

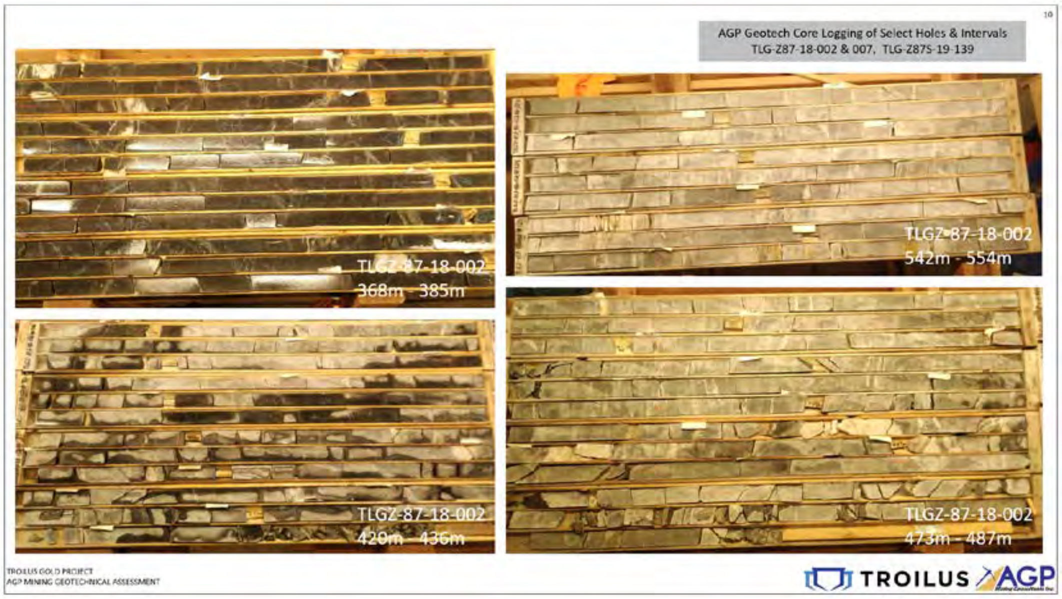
- meetings with Troilus exploration geology staff
- technical presentations regarding local & regional geology
- discussion & review of past and current exploration drill plans and status
- inspection of core cutting, sampling, and logging facilities, discussions with logging technicians (Figure 16-1)

Figure 16-1: Troilus Core Logging Area

- review of previous geotechnical reports and data (incl. on-file paper documents provided by Troilus), incl.:
    - geotechnical design and production reports for the open pit & preliminary u/g mining studies
    - open pit geology & preliminary (basic) geotechnical databases, drill hole locations, hole orientations
    - oriented structural measurements from drill core and mapping
    - 3D geology & structure model & data
    - laboratory test data, field test data
    - (limited) historic pit dewatering information and records
    - pit photos / core photos
    - historic Inmet open pit production information and mining data, including document file folders and >20 CD-ROM project files from 2001-2005

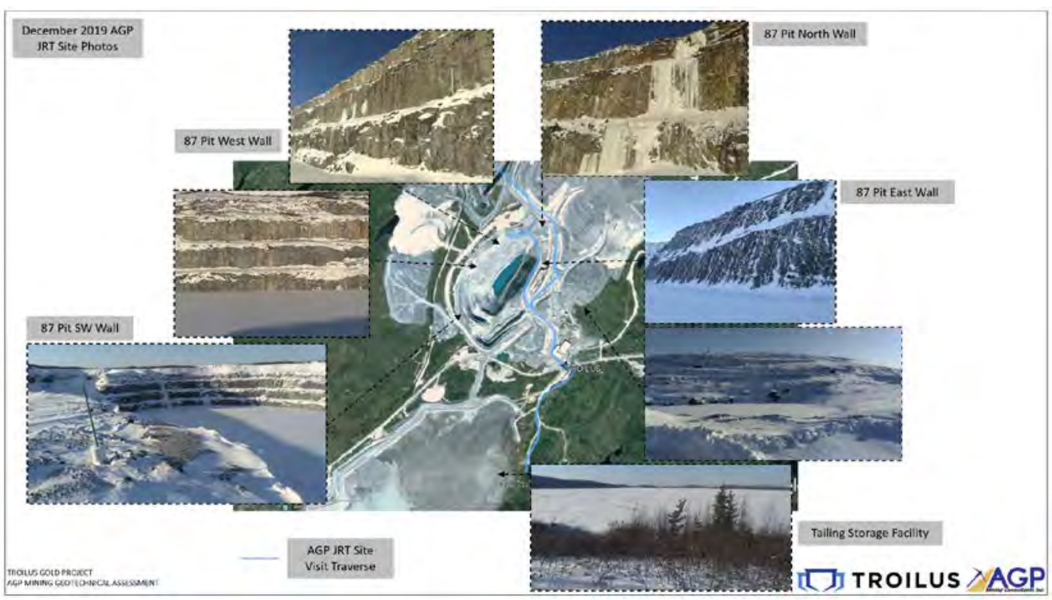
- geotechnical logging of select / key holes and intervals from available core (TLG-Z87-18-002 & 007, TLG-Z87S-19-139) (Figure 16-2)

Figure 16-2: AGP Logged Core



- snow machine traversing of accessible pit slopes, access roads, and tailings storage facility (Figure 16-3)

Figure 16-3: Site Visit Photos



- geotechnical mapping and rock mass characterization of outcrop rock masses, focused on verifying and supplementing existing information, including lithology, rock mass strength, and discontinuity characteristics.; as recorded by others, overall, pit walls in both the Z87 and J4 pits have been constructed in 'Good to Very Good' rock and continue to exhibit overall stable conditions

16.2.2 Project Mining Geotechnics

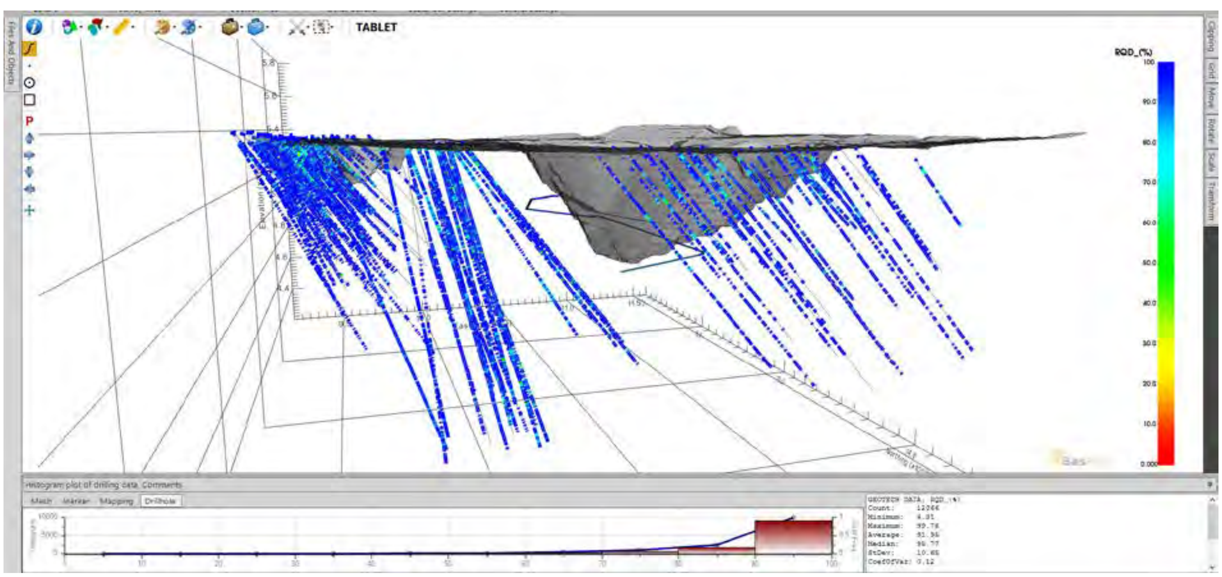
Geotechnical knowledge for this PEA benefits significantly from previous operating experience and related project investigations, analyses, and mine designs. Project geology and geotechnical units have been well defined by others (Piteau (1996-2006), Bawden (2005), Itasca (2006), SRK (2018)) and further confirmed by AGP for the current study.

A number of parameters have been considered by AGP in confirming a preliminary geotechnical model for the project; including available information on lithology and geologic structures (detailed in Section 7), core recovery and rock quality data (RQD), intact rock strength data (from previous field index testing and limited laboratory testing), and joint spacing, condition, and infill characteristics, as well as related rock mass classification data.

Geotechnical observations and data indicate generally 'Good to Very Good' rock mass conditions throughout the mining zones [i.e. throughout the hanging wall, ore zone, and foot wall units].

RQD data typically ranges from zero, in upper hole intervals and fault zones, to 75% to 90%+ in most drill runs. Historically recorded RQD values have been reviewed, error-checked, and plotted by AGP along drill hole surveys to assist in developing a three-dimensional approximation of the condition and variability of rock quality in the vicinity of the proposed pit slopes and underground mine workings. Views of these data are presented in Figure 16-4. Preliminary Rock Mass Classification models using available data have also been developed using similar techniques.

Figure 16-4: Troilus Project RQD Values by Drill Hole



Intact rock strengths have been determined from previous (limited) laboratory testing and geotechnical core logging (Table 16-1).

Table 16-1: Intact Rock Strength

Rock Type	All Valid Tests	Metadiorite/Diorite Breccia	Felsic Volcanics
Population	49	34	20
Average UCS (MPa)	160.8	152.4	171.2
Standard Deviation (σ) (MPa)	50.7	50.1	44.5
Average UCS = 1xσ (MPa)	110.1	102.3	126.7
Median (MPa)	150.6	145.0	150.6
20% Cumulative Frequency (MPa)[6]	113.0	101.2	138.7

Geotechnical logging has recorded typical rock strengths ranging from 100 to 200 MPa (using the widely accepted ISRM RO to R6 scale).

Typical discontinuity spacing observed on bench faces and in drill core varies from 0.2 to 0.6 m. Discontinuity surfaces vary from mostly slightly-rough to smooth to slicken sided with minor amounts of soft clayey infill, 0 - 5 mm thick.

As described in Section 7, 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 zones in the Z87 pit. Faulting is also 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.

Limited discontinuity strength test data currently exists. Consequently, conservative estimates have been made for ranges of strengths, based on a review of the qualitative data and relevant experience in similar rock masses. For this PEA joint strengths have been estimated between 25 – 35 degrees friction, with faults likely between 15 – 25 degrees, both with zero to nominal cohesion.

Geotechnical units have been previously defined by others (Piteau (1996-2006), Bawden (2005), Itasca (2006), SRK (2018)) and further confirmed by AGP for the current study. Experience indicates these units are more appropriately based on observed ranges in rock mass classification values, rather than, specific lithological or structural domains. There appears to be limited variation between rock mass classification values estimated for the footwall, ore zone, and hangingwall rock masses. In terms of spatial distribution, values appear to be slightly lower towards the north end of the deposit, which may reflect the proximity of the hangingwall felsic dyke to the ore zone in this area. Previous estimates of RMR values typically range from 60 to 80+ for all geotechnical units; these ranges have again been used by AGP in the current study (Table 16-2).

Table 16-2: Geotechnical Model Framework (Piteau, 2006)

Zone	Average RMR_{76}	Min RMR_{76}	Max RMR_{76}
Footwall / Hangingwall	71	60	82
Ore Zone	69	60	85

These RMR ranges have also been used to estimate rock mass strength and deformation parameters for subsequent analysis and design tasks. Related Mohr-Coulomb and Hoek-Brown strength envelopes have been estimated over stresses that are a function of the proposed slope heights and underground mining depths. Conservative "Disturbance Factors" have been assumed in deriving the various rock mass strength parameters, indicative of significant disturbance to the rock mass due to production blasting and /or local stress redistributions resulting from mining activities.

16.2.3 Open Pit Mining Geotechnical

Initial estimates of suitable pit slope angles for PEA-level mine planning have been determined based primarily on field observations, records from previous mining operations, and previous mining-geotechnical design reports completed by Inmet and their design consultants between 1996 and 2006. Recent data and information collected from on-going resource drilling and core logging by Troilus, including core recovery and rock quality designation (RQD) data, has also been incorporated. No 'targeted' geotechnical drilling, logging, mapping, sampling, or laboratory testing was completed for the current study.

Per Inmet (2003) the original feasibility study design for the 87 Pit included a 55° inter-ramp slope angle for both the main East and West walls (Roche, 1991 and Supportek, 1993). Initial production upgrades by Inmet using pre-split blasting techniques obtained superior results, permitting inter-ramp angles of up to 59° for the West wall. Following some rock instability problems after the first few benches were excavated in the East wall, a new design was incorporated to resolve stability issues related to joints that were parallel to the foliation (Piteau, 1997). Further structural mapping studies recommended an inter-ramp angle of 52.5° using 20 m benches and 8 m safety berms (Piteau, 2001).

The original design for the North and South walls was also based on a 55° inter-ramp angle. After structural mapping in 2000 and 2001, the North and South wall angles were increased from 55° to 59° by reducing the safety berm width from 14 m to 12 m (Piteau 2001).

During the 2002 J4 definition drilling campaign, Inmet identified groups of fractures with the help of orientated drill core. Fracture analysis showed the same systems as for the Z87 pit (Bélanger 2002). The rock competency in the J4 zone was recorded as excellent and the rock quality designation (RQD) was observed above 90%. In addition to the fracture analysis study, a second study was undertaken to identify and correlate the main fault structures present at J4 (Bélanger 2002).

As demonstrated above, AGP notes efforts to 'maximize' slope angles at the early stages of a project based on limited *location specific* data can often lead to overly optimistic designs and related project economics. These can be difficult to 'walk back' if, or when, contrarian data is recorded during subsequent investigation work or worse, during excavation. AGP recognizes the potential to optimize relatively conservative initial guidance, if/when additional confirmatory data becomes available.

The overall slope angles for use in LG routines (Section 16.3) are shown in Table 16-3. Slopes were flattened as required due to inclusion of haulage ramps.

Table 16-3: Pit Shell Slopes

Pit	Azimuth (degrees)	Material Type	Overall Slope (degrees)	Description
J/Z87	**0-360**	**Overburden**	**30**	
	30	Bedrock	49.2	One 33.2m wide ramp, IRA = 53 deg, wall height of 300m
	45	Bedrock	43.8	One 33.2m wide ramp, IRA = 47 deg, wall height of 300m
	135	Bedrock	43.8	One 33.2m wide ramp, IRA = 47 deg, wall height of 300m
	150	Bedrock	49.2	One 33.2m wide ramp, IRA = 53 deg, wall height of 300m
SW	**0-360**	**Overburden**	**30**	
	30	Bedrock	48.3	One 33.2m wide ramp, IRA = 53 deg, wall height of 240m
	45	Bedrock	43	One 33.2m wide ramp, IRA = 47 deg, wall height of 240m
	135	Bedrock	43	One 33.2m wide ramp, IRA = 47 deg, wall height of 240m
	150	Bedrock	48.3	One 33.2m wide ramp, IRA = 53 deg, wall height of 240m

Observations and mining experience confirm the noted criteria are practical estimates of achievable slope configurations. If the project is feasible at these inter-ramp slope angles, any improvements or further optimization that can be achieved as a result of additional geotechnical study for given scenarios will be value-additive. If on the other hand project economics are marginal when these criteria are incorporated, this simply highlights the necessity of determining the project's geotechnical conditions as soon as practical.

The following widely used empirical slope stability charts (Figure 16-5 and Figure 16-6) demonstrate typical safety factors for a variety of slope configurations and rock types and include AGP's preliminary guidance.

Figure 16-5: Slope Height by Slope Angle – Factor of Safety (Hoek & Bray, 1974)

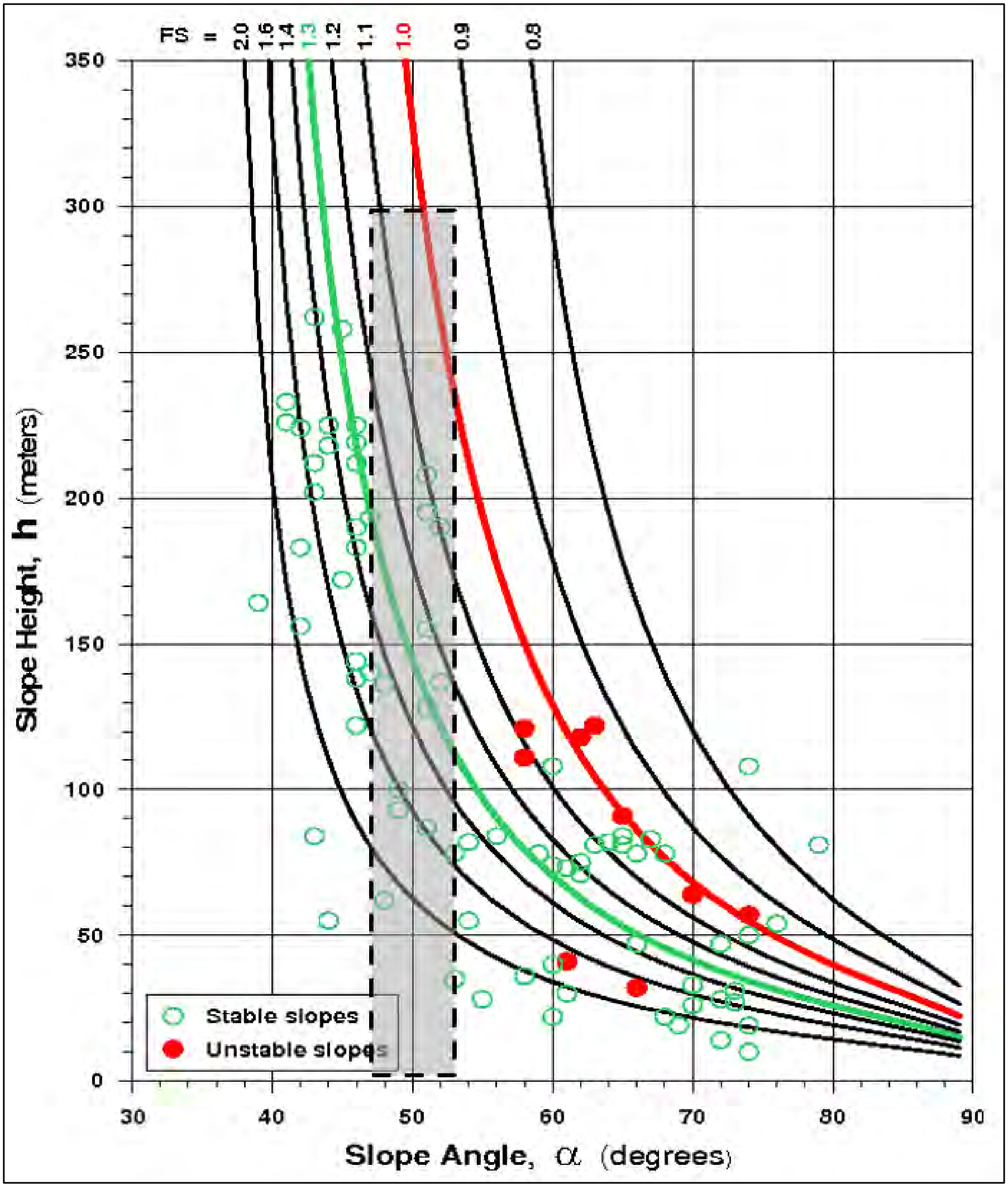
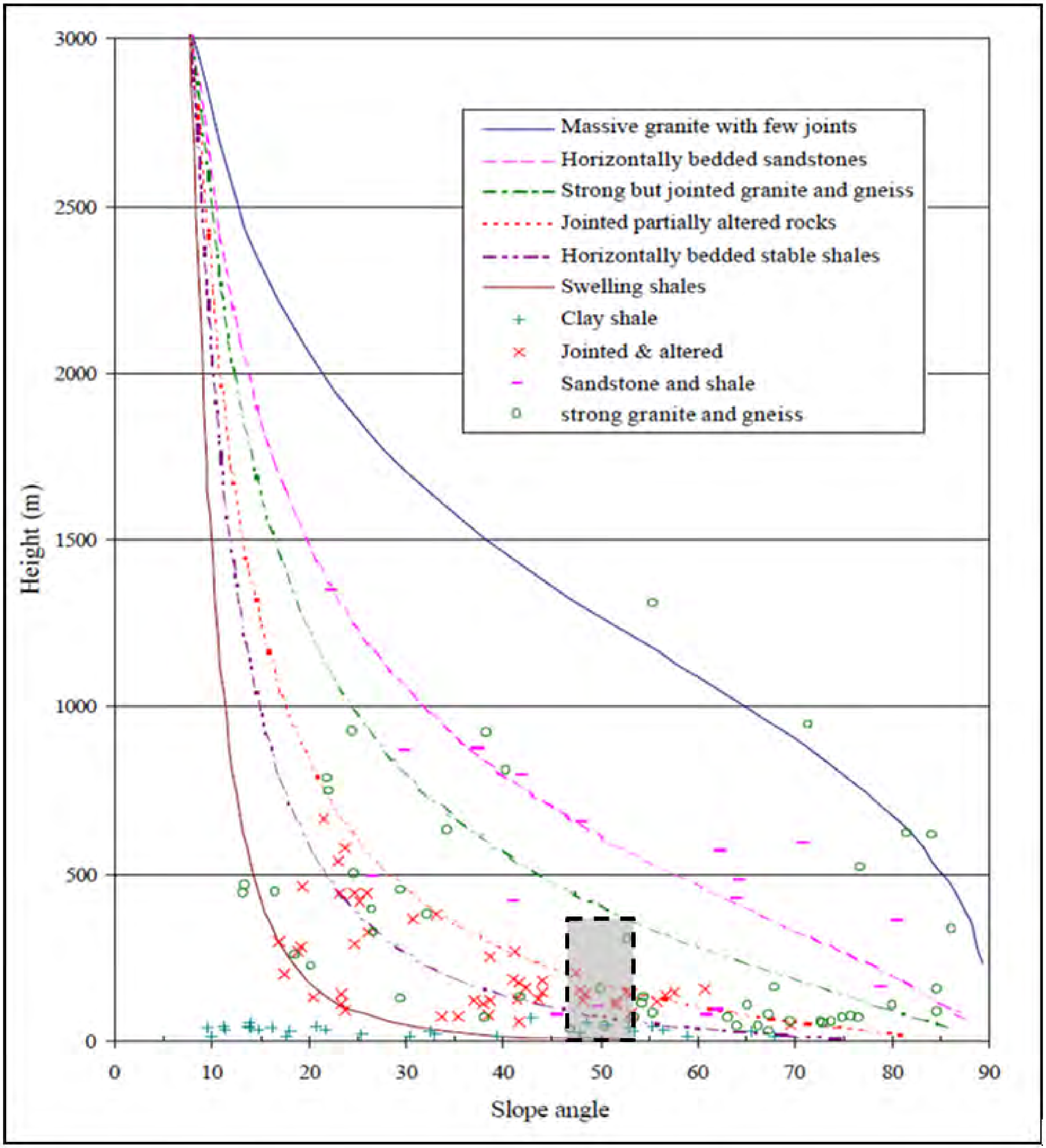


Figure 16-6: Slope Angle by Slope Height – by Rock Type



Preliminary 2D limit equilibrium and 3D FEM and BE analyses (Figure 16-7 and Figure 16-8) were completed by AGP to gather initial insight into inter-ramp and overall slope geotechnics. While currently generic in nature, these models have been used to assess and interpret a wide variety of geotechnical and slope stability issues that may arise as a result of future investigations, including changes to the geological and structural models, variable non-linear and anisotropic rock mass strength criteria, evolving stress conditions at the open pit / underground interface, and ground water conditions.

AGP commonly uses the following approach for target Factor of Safety (FOS) values at the PEA level of study:

- multi-bench or inter-ramp slopes controlled by discontinuities should achieve a minimum FOS of 1.2
- inter-ramp or overall slopes involving shearing through the rock mass and with a low or medium consequence of instability should meet a minimum FOS of 1.3
- overall slopes with a high consequence of instability should meet a minimum FOS of 1.5

Figure 16-7: Limit Equilibrium – 2D

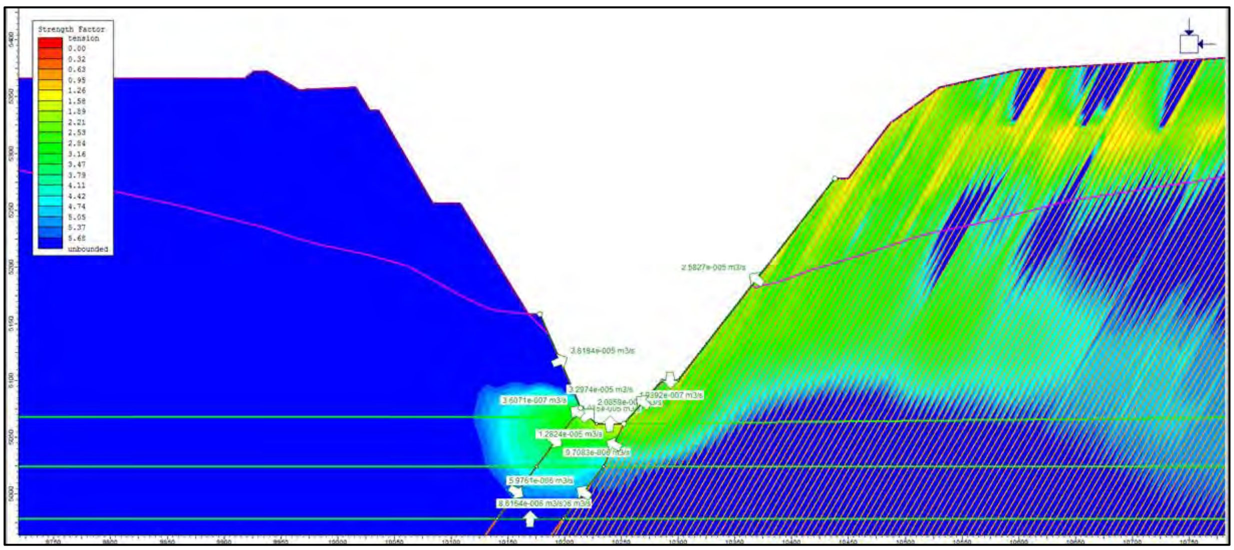
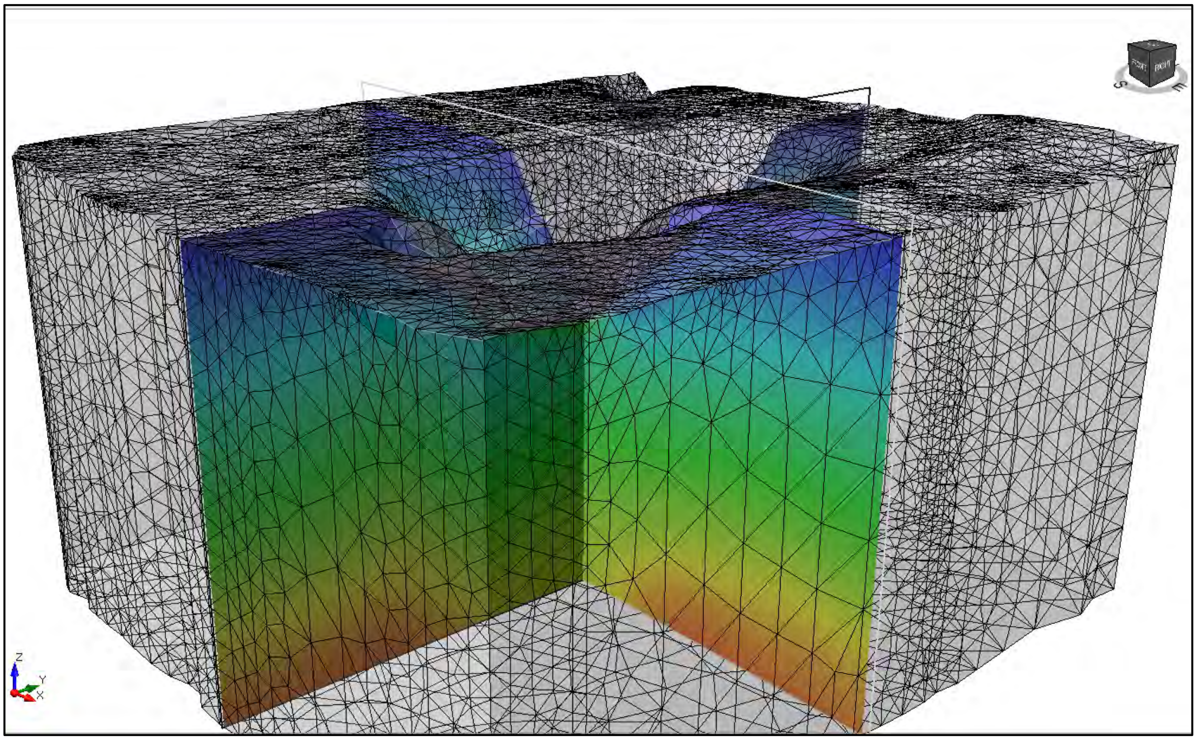


Figure 16-8: Finite Element Mesh – 3D FEM



The preliminary analyses indicate the ‘as-designed’ slopes are predicted to exhibit generally ‘stable’ conditions for a variety of scenarios, with typical ‘minimum‘ FOS’s ranging from ~1.2 - >2.0 for inter-ramp and global slopes.

Bench scale slope instabilities have not been assessed in detail for the current study. Bench configurations have included an allowance for reasonable catchment widths to help manage operational challenges that may arise from local bench-scale stabilities resulting from adversely oriented geologic structure. It is known from previous experience that adversely oriented geologic structures are present locally within various slope pit sectors (partially portions of the SW and SE walls); it is assumed at present that small bench-scale failures developed along these features can be managed with careful blasting techniques and regular berm maintenance/clearing, wherever access is possible. As an alternative, it is notable that Inmet successfully used limited / local bench rock support during previous operations.

AGP notes seismic loading and multi-bench-scale to pit-scale geologic structures have the potential to significantly affect overall pit slope stability. The current status and impact of these are both largely unknown and will need to be determined at the next level of study. The inclusion of hypothetical adversely oriented faults and bedding planes in the stability analyses indicates potential FOS’s less than 1.0, particularly with various seismic loads applied. Further geotechnical investigations are warranted to assess for these as-yet undetected / unlikely structures, and the location and character of any inter-ramp to global-scale features that may impact stability and mining outcomes.

Sufficient data has been compiled regarding geotechnical strengths and characteristics of the primary rock types to support the range of potential pit wall design guidelines provided. However, numerous assumptions had to be made about the primary controls on rock mass stability, geology, rock mass strength, groundwater pressures, and potential failure mechanisms. As such, the stability models should be considered conceptual in nature. Updated models at that address these uncertainties are required at the next level of study.

Underground Mining Geotechnical

In contrast to the significant operating experience used to inform the above noted open pit slope designs, less reliable information is currently available for underground mine designs. As such the current underground assessment is appropriately characterized as a 'PEA-level assessment' (i.e. an assessment of mining-geotechnics as it relates to the reasonable prospect for eventual economic extraction of mill feed at Troilus).

AGPs assessment is based on a limited review and analyses of available data from the underground zone, and previous underground mining studies and plans by various subject matter experts (Bawden, 2005, Itasca, 2005, Inmet 2006, Piteau 2006, SRK 2006, RPA 2018). The current project is understandably significantly less advanced than 'Feasibility Level' assessments and related stress modeling studies completed by Inmet and their consultants during previous operations, and for previous project iterations.

For the current PEA AGP sought to merge the previous studies, stress modeling, and mining experience, with available geotechnical data, to inform the proposed mining methods and to address what may be broadly 'probable' at Troilus, with the understanding that further work will be required to verify the concepts and designs presented This assessment is not intended to be a "conservative" or "low-risk" assessment.

The Troilus deposit is a large, low grade resource. Previous studies have indicated high grade, selective, and high cost mining methods are inappropriate for this deposit. High tonnage, low cost mining methods offer the only economically attractive solutions (Inmet 2006).

As will be described in further detail in Section 16.4, for the current study a number of differing underground mass mining methods were initially assessed for suitability for application to the low grade deposit at Troilus From this trade off study "Slot and Mass Blast" (S+MB) with introduced waste material emerged as the preferred method. Slot and Mass Blast attempts to balance necessary high tonnage production requirements with the geotechnical constraints and opportunities that are present at Troilus.

S+MB is a top-down non-entry high tonnage stoping method. Primary 'Slot' excavations are intended to remain 'stable' and open only temporarily. The remaining rib pillar and crown pillar are subsequently fired as a single "Mass Blast". The dimensions of the slot stopes and crown and rib pillar are designed such that the open stope provides an opening sufficient to accommodate the swelling of the material broken in the mass blast with the level of the broken mass blast mill feed at roughly the top of the crown pillar elevation, while providing stabilizing confinement to the excavation walls.

Stopes, and temporary ribs, and crowns are sized to remain stable as required by the mine sequence, and limit the extent and potential for local zones of geotechnical weakness to regularly coalesce into multi-level regional failures; and within the HW to limit ‘bridging’ failures and progressive regional dilutive caving of the rock mass, at least until the stope has been largely pulled of mill feed and abandoned to waste.

S+MB is distinct from previously assessed Long Hole Open Stoping (LHOS) methods in that it relies on the stopes remining full of broken rock under controlled draw conditions once the secondary “Mass Blast” is taken. Empirical conceptual stability analyses and stress modeling described below indicates S+MB is a plausible methodology at Troilus, with acknowledged risks and uncertainties.

Conceptual and modeled S+MB dimensions considered for this study are as follows:

- Slot Dimensions
  - full orebody width from hangingwall to footwall (average ~ 37 m)
  - 30 m along strike
  - 60 m tall
  - hanging wall HR = 10 / Back HR = 8.5
- Mass Blast Dimensions:
  - full orebody width from hangingwall to footwall (average ~ 37 m)
  - 30 m crown pillar height
  - dimension along strike variable to allow mass blast muck to fill the full void created by both the slot and mass blast portions of the stope. This along strike dimension averages 35 m; for the purposes of this PEA, a fixed mass blast strike dimension of 35m was used for all stopes

The upper most stopes (mined as a sub-set, using SLC methods, as described in 16.4) are to be drilled and blasted through to the bottom of the pit. Prior to the first mass blast open pit waste will be dumped or placed in the base of the open pit. There will inevitably be some mixing of mill feed and waste and broken material will be pulled to a draw point cut-off grade. AGP has assessed that the uneconomic hanging wall rock mass will generally be too strong to cave predictably and in substantial tonnages during local drawdown, particularly with stope walls fully confined by blasted ore. This requires further investigation at the next level of study. Draw control, and stress and mine sequence management become key controlling geotechnical parameters for mining execution.

As concluded by Itasca, Inmet and others, and confirmed by AGP, there is no doubt that the proposed excavations contemplated by the various mining methods (both open stopes in the past, and waste-filled presently) are ‘large’ by normal underground mining standards. Inmet and Itasca’s 2006 work and conclusions contribute relevant perspective: Itasca concluded open stopes up 265 m high (from 4770 m level to 5035 m level) were potentially possible, likely exhibiting general stability with acknowledged dilution potential and related risks and uncertainties(Figure 16-9 and Figure 16-10).

Figure 16-9: Itasca (2006) Underground Mining Concept – Option IV

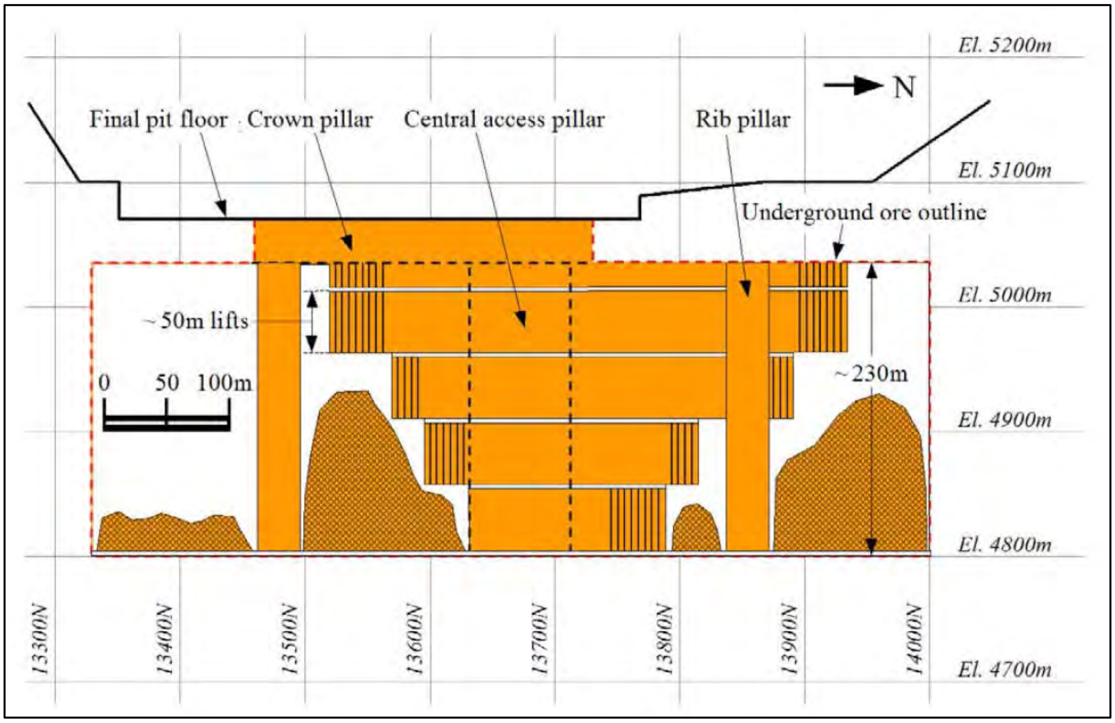
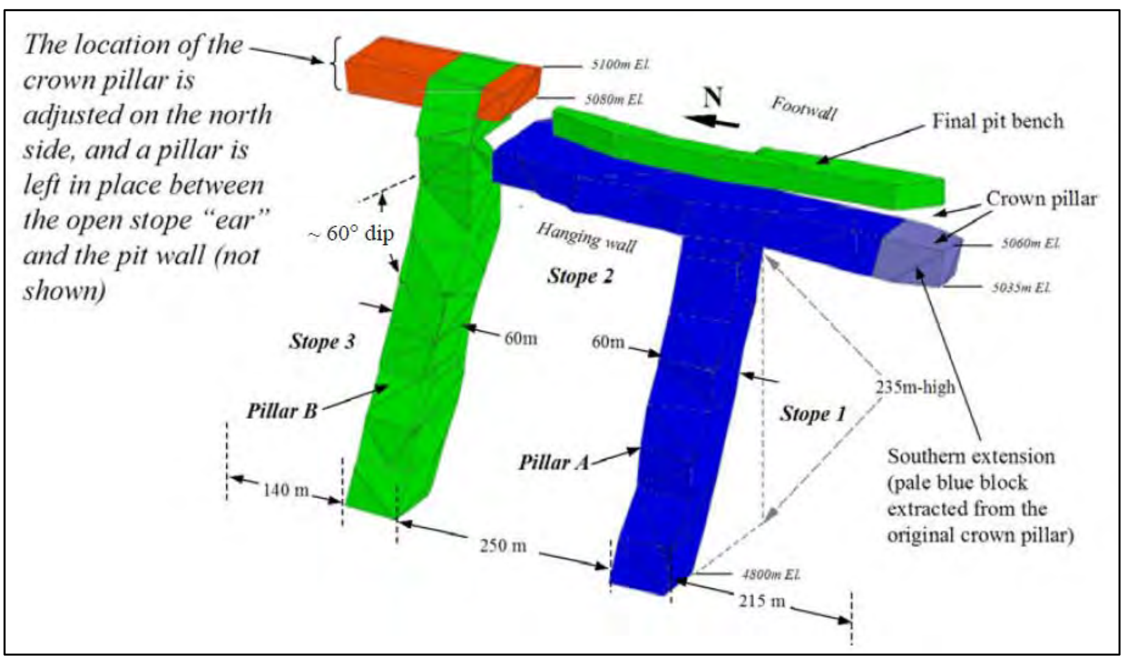


Figure 16-10: Itasca (2006) – Underground Mining 3DEC Simulation – Option 10



Inmet reduced these conceptual dimensions somewhat following concerns raised by other subject matter exerts. However previous and current analyses completed with the widely-used 'Empirical Stability Graph' and related 'ELOS Dilution Graph' methodologies nonetheless recognize the potential for the extraction of large-scale, highly product stopes, with permissible Hydraulic Radii for key back and hanging wall geometries consistent with those contemplated for current designs. It is noted that for a moderate percentage of likely geotechnical and geometric conditions, the stability of largest proposed stopes and pillars is uncertain-to-unlikely (see industry standard Modified Stability Graph charts, below); These predicted instability and ELOS dilution potentials (applied as described in Section 16.4) may be locally mitigated to some extent with deeply anchored rings of ground support installed from development drives or alternatively from local sub-drives, as may be required. It is notable that there is very limited data available in the Stability Graph for very large excavations, which limits its applicability for such conditions (Figure 16-11 to Figure 16-13).

Figure 16-11: Stability Graph Method (Bawden 2005)

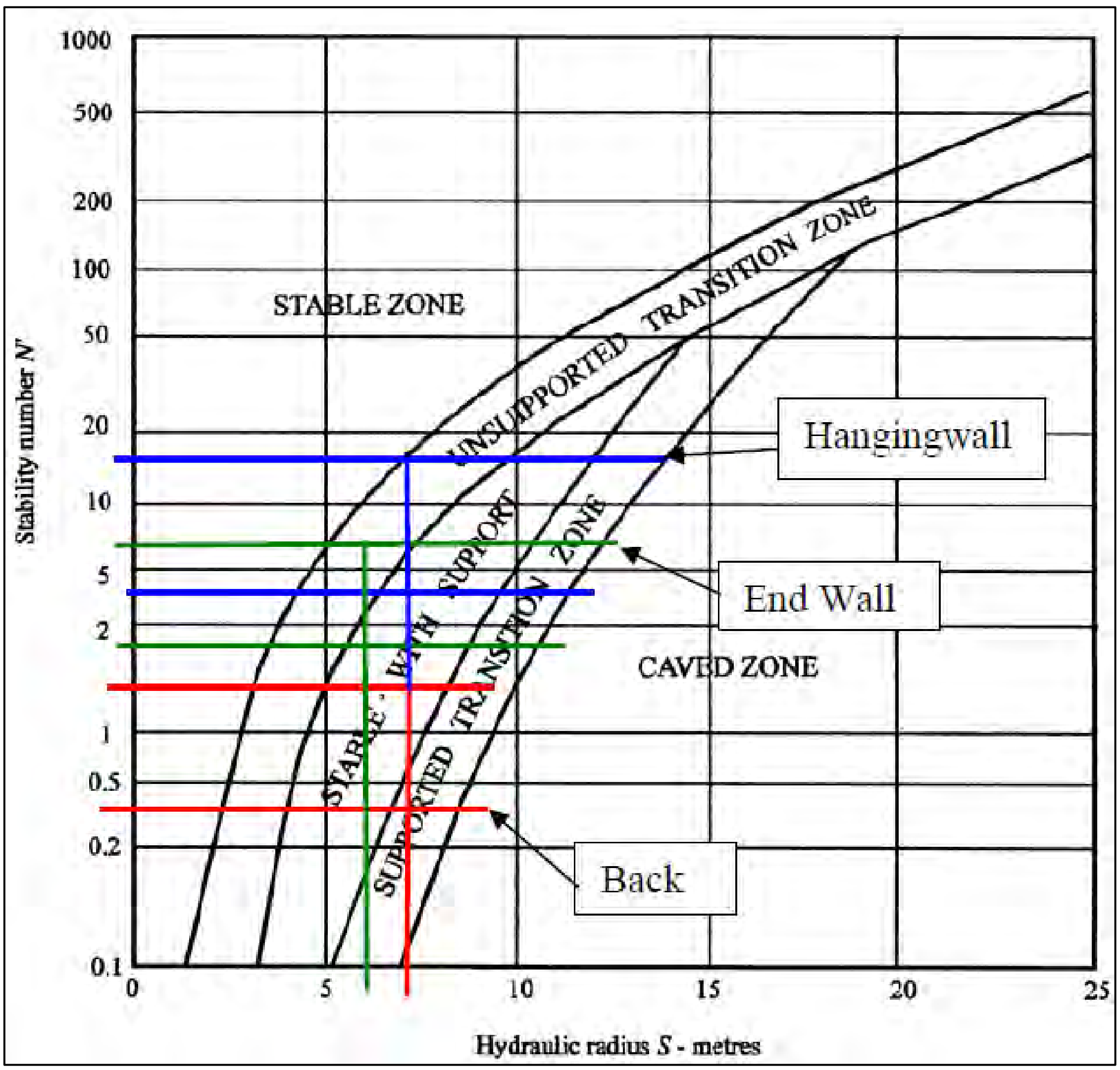


Figure 16-12: Modified Stability Graph Method (Itasca 2006)

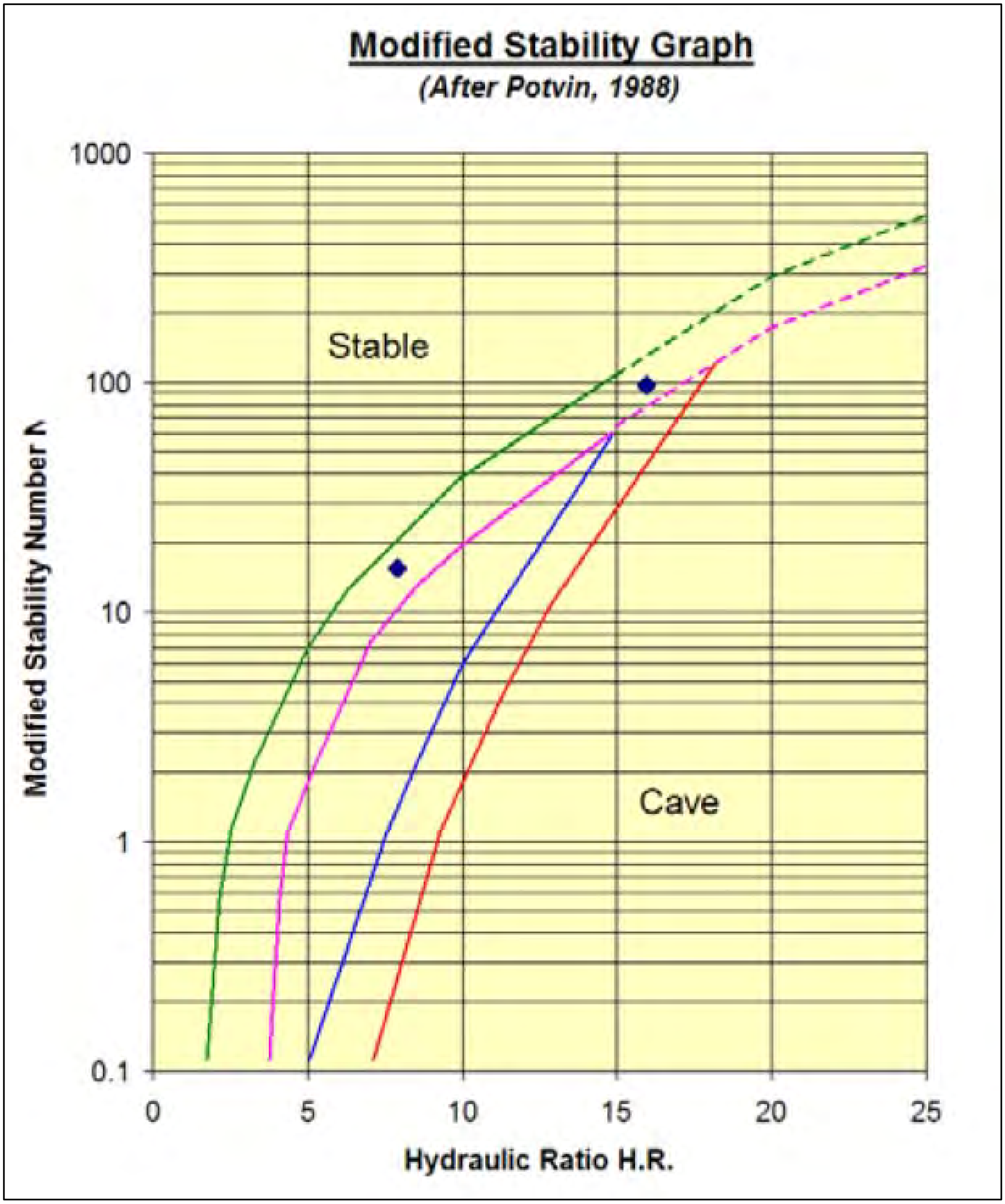


Figure 16-13: Modified Stability Graph Method (RPA 2018)

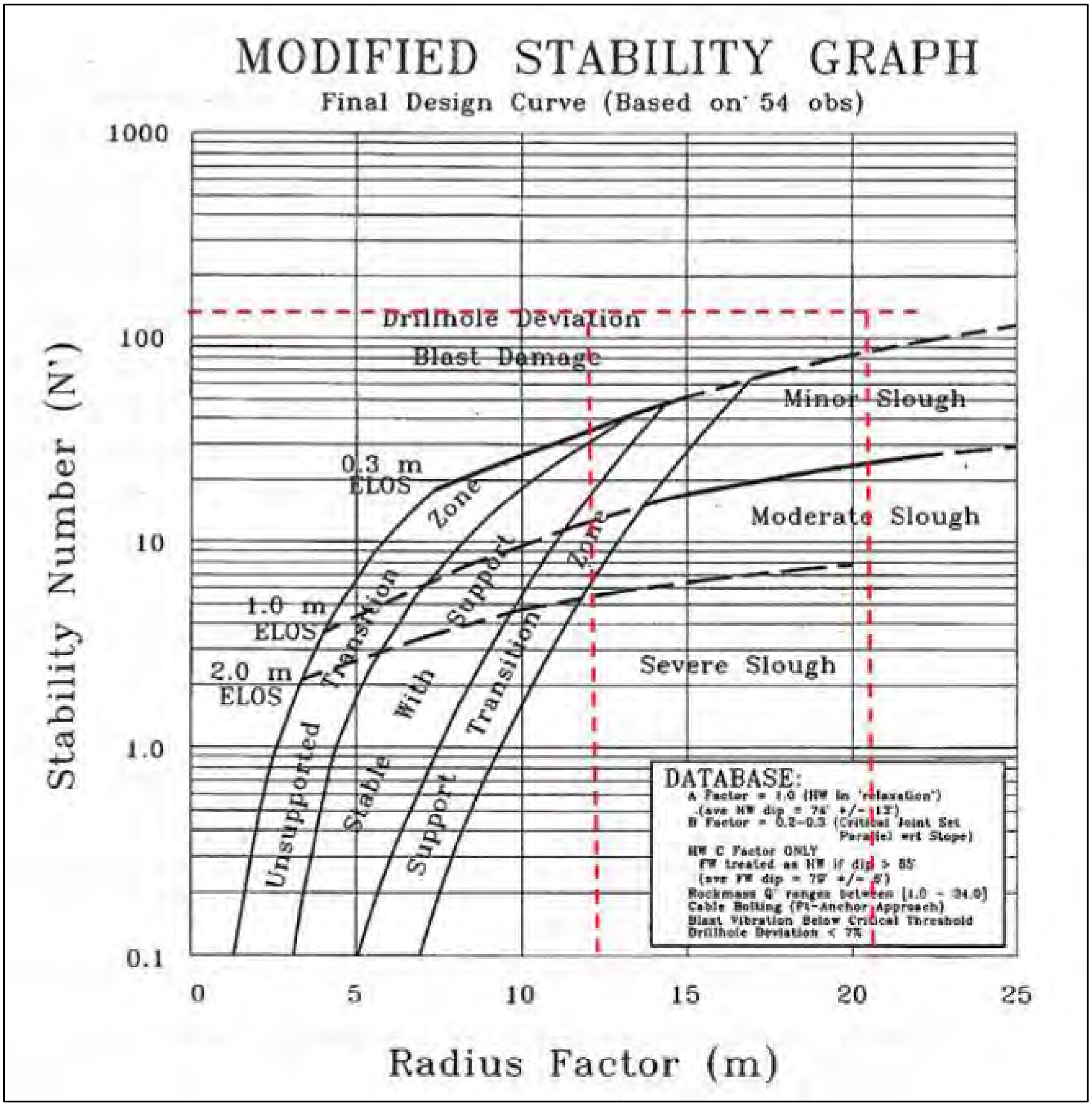
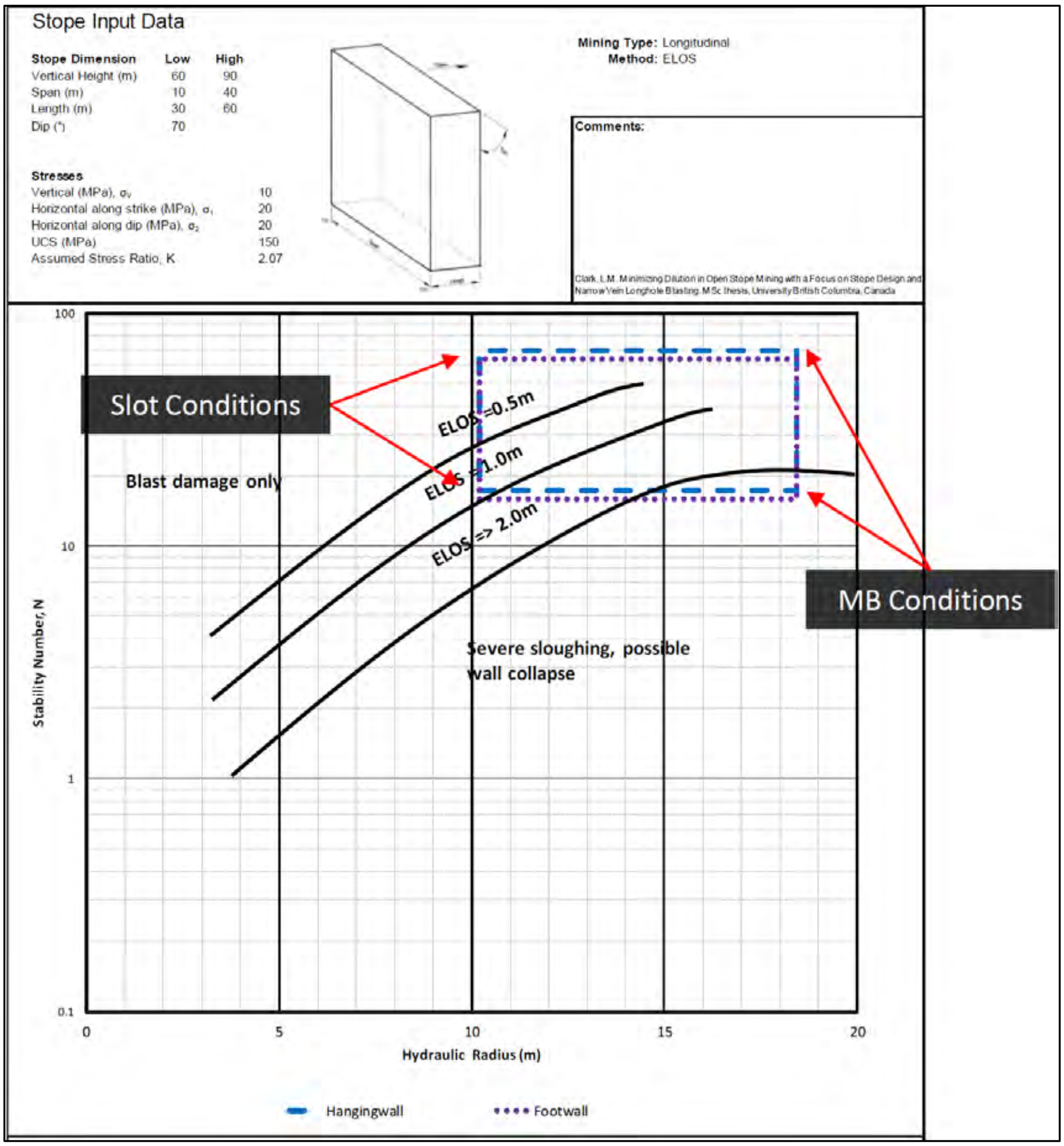


Figure 16-14 indicates the conceptual dilution estimate for Slot and Mass Blast conditions. These have been considered when applying mining dilution estimates, as described in Section 16.4.

Key reasons both professional experience and confirmatory stress analyses indicate that large openings may be stable at Troilus are:

1. strong rock mass properties and the assumption of a low to moderate stress field
2. a jointing system that is for the most part is favorably oriented

Figure 16-14: S+MB ELOS Dilution Graph Method (AGP 2020)



Previous and current 2D and 3D Numerical Models (Figure 16-15 to Figure 16-18) have been used to explicitly simulate the evolution of stresses within the rock mass as the proposed mining sequences progress, and more ground is incrementally removed. The numerical analyses indicate that it is not unrealistic to target high mill feed recovery levels (in the 80% range) using large open stopes, and slots in the case of S+MB. Both continuous and jointed analyses have reached this conclusion.

Figure 16-15: 3DEC Numerical Model Results – Dilution Assessment (Itasca 2006)

Geomechanics Assessment With 3DEC Numerical Analyses of Underground Stoping Options at the Troilus Division. *FINAL VERSION* Andrieux, O'Connor & Turichshev *January 2006.*

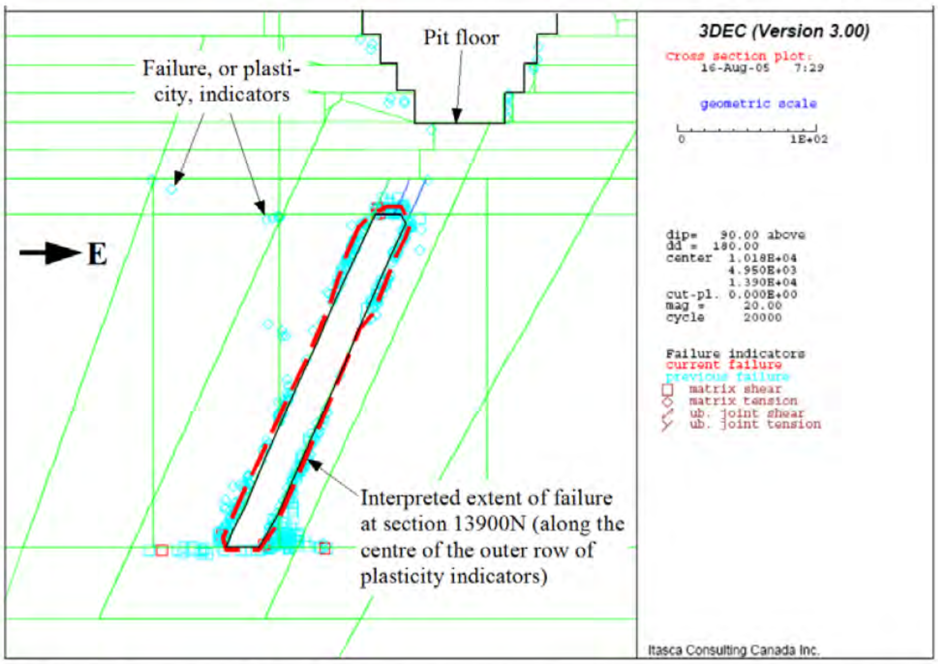


Figure 12. Second stage of a dilution assessment. Example assessment on a cross-section cut at 13900N of the extent of material expected to end up in the stope, as dilution.

Geotechnical data has demonstrated that there is limited variability in the rock mass quality and intact strength parameters of the different lithologies found at Troilus. The minor differences between the intact rock strengths and rock mass classifications have resulted in similar material properties. Therefore, the numerical modeling completed for the project has typically incorporated a single material domain, with ranges in values assessed through sensitivity analyses of the various strength inputs and in situ loading conditions.

AGP's preliminary stress analyses of the application of S+MB method at Troilus has highlighted predictable 'areas of interest' including stress concentrations and zones of stress shedding and relaxation, along with potentially adverse conditions in 'crown pillars' & top drives after slot blasts, and multi-level / late-stage de-stressing of hanging walls; however, for the purposes of this PEA, the models generally exhibit stability conditions demonstrating a 'reasonable prospect for extraction'. This information, along with basic geotechnical data described above has been used to develop preliminary ground support guidance for preliminary cost estimating purposes, and define infrastructure development offsets, as further described in Section 16.4.

Modeled rock mass regions with induced stresses approaching 1/3 to 1/2 of UCS values and Strength Factor's (SF) less than 1 are minimal to moderate and localized, and generally predictable; given the high intact rock strengths the model results confirm the probable limited extent of rock mass yield, with the expectation that these conditions can evolve significantly as mining depths are increased. (Figure 16-16, Figure 16-17, Figure 16-18).

Figure 16-16: Modelled Rock Mass – View 1

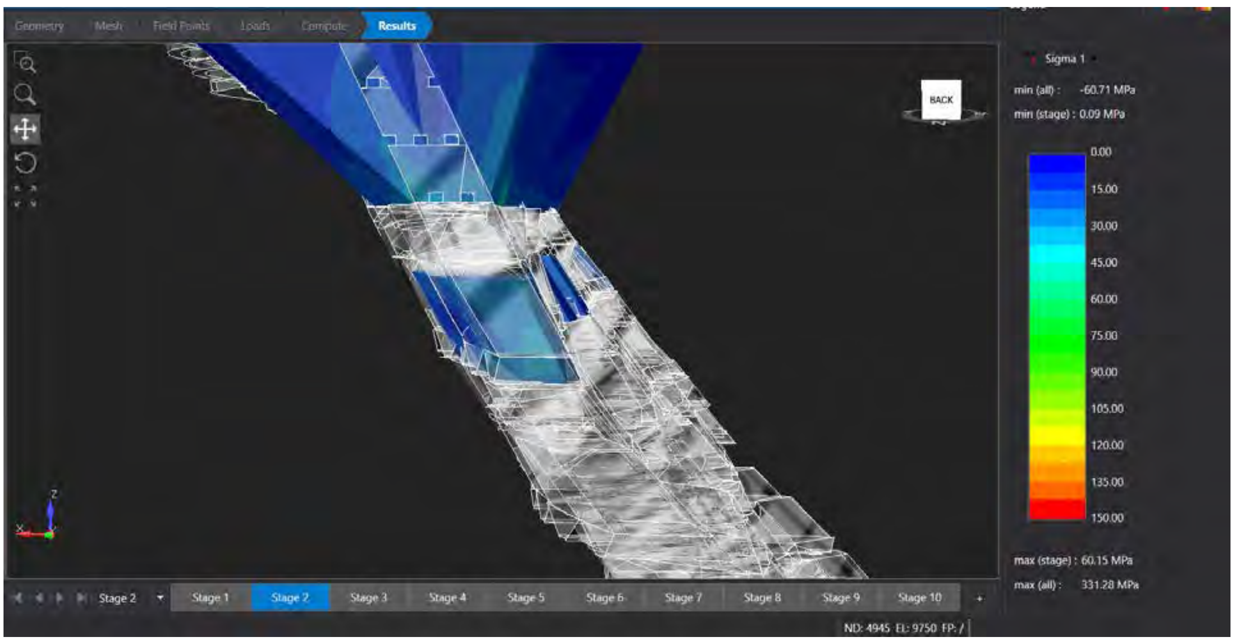


Figure 16-17: Modelled Rock Mass – View 2

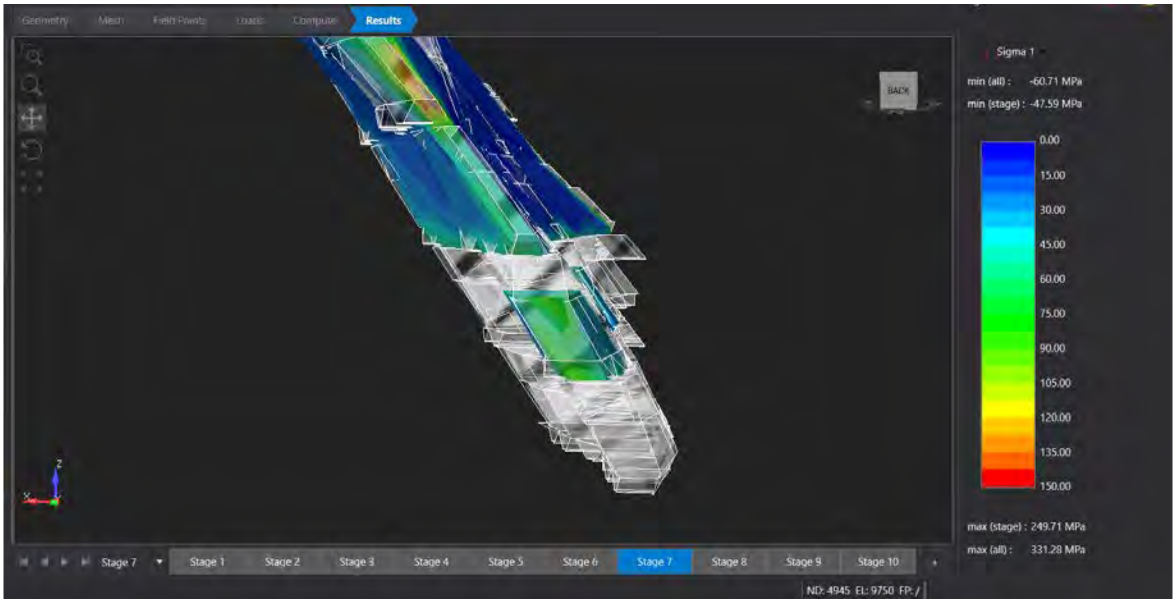
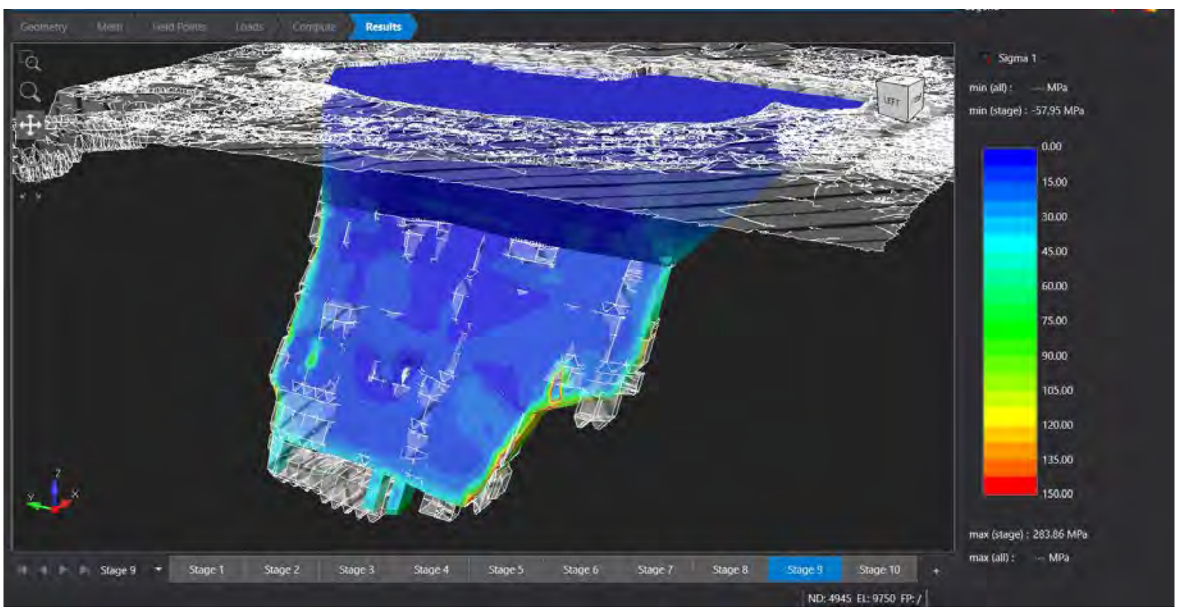


Figure 16-18: Modelled Rock Mass – View 3



The risk of encountering adverse features and stress conditions increases with the size of the mining-disturbed rock mass and mining depth. Considering the dimensions contemplated at Troilus, it is reasonable to expect that at least some adverse conditions will be encountered on each stoping level. Work should continue to characterize the rock mass as accurately as practical. Of particular importance will be the detection and precise delineation underground of faults, dykes, zones of weaker rock and any other weakening feature in the rock mass. It is well recognized that positive experience and observations from mining 87 Pit reduces the level of uncertainty to some degree.

AGP's proposed Slot and Mass Blast is a thoughtful and industry-proven conceptual option that attempts to balance necessary high tonnage production requirements with geotechnical constraints and opportunities at Troilus. The S+MB method proposed by AGP is distinct from other LHOS methods in that S+MB method relies on the stopes remining full of broken rock under controlled draw conditions once the Mass Blast is taken. Empirical and preliminary stress analyses completed by various experts and confirmed by AGP suggest largely stable production conditions for this configuration. Importantly, stope positions and widths can conceivable be moderately adjusted on the individual level to de-risk / optimize local geometries with the collection of improved geotechnical data.

16.2.4 Hydrogeology

Existing mine hydro-geology information and data was reviewed for the current PEA to estimate conceptual inflow rates for open pit and underground mining zones.

The data included regularly collected precipitation readings taken throughout previous operations (Table 16-4), historic hydraulic conductivity testing and water level measurements from boreholes within and proximal to the mining zones, and mine dewatering rates and pumping information collected by Inmet. More recent hydro-geology data including information contained in a 2008 Genivar Study and Troilus's 2019 87 Pit dewatering evaluation were also incorporated.

Table 16-4: Historic Average Water Infiltration Rates – 87 Pit

Month	1997	1998	1999	2000
(guspm)				
January	266	799	592	918
February	266	1110	533	876
March	651	1024	1024	1332
April	1983	987	1169	1103
May	947	592	1006	1154
June	148	691	1088	
July	148	1016	1117	
August	118	1006	953	
September	178	518	984	
October		688	1886	
November		318	1219	
December		607	1095	
Average	**534**	**788**	**1052**	**1077**
m^3/h				
January	60.3	181.2	134.3	208.2
February	60.3	251.7	120.9	198.7
March	147.6	232.2	232.2	302.1
April	449.7	223.9	265.1	250.2
May	214.8	134.3	228.2	261.7
June	33.6	156.7	246.8	
July	33.6	230.4	253.3	
August	26.8	228.2	216.1	
September	40.4	117.5	223.2	
October		156.0	427.7	
November		72.1	276.5	
December		137.7	248.3	
Average	**121.1**	**178.7**	**238.6**	**244.3**

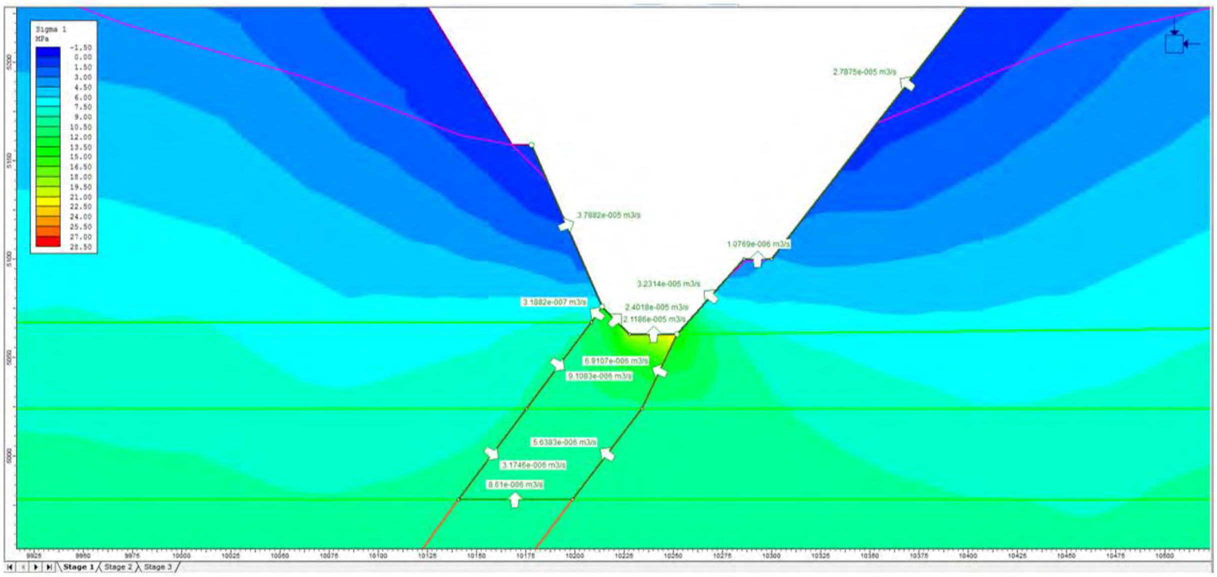
Note: Troilus surface mine was in operation from November 1996 to April 2009

In an internal memo (Inmet, September 18, 2001), reference is made to a water inlet in the bottom of 87 Pit varying between 1,200 and 1,500 usgpm (75 - 95 l/s or 270 – 340 m3/hr), 50% of which would come from South and South-East sectors, 40% of the North sector and 10% of the East and West walls.

The hourly average flow rate of the combined effluent from pits 87 and J4 would have been 338 m3 / h in 2006 and 275 m3 / h in 2007 (Genivar, 2008). The dewatering rate seems to have increased to the rate of about 340 m3 / h, then stabilized and even decreased significantly. This suggests that a state of equilibrium was reached between 2001 and 2006.

From the above data order of magnitude inflow rates have been estimated for the open pit and underground mining zones (see Section 16.4), in consideration of the mining schedule and seasonal variations, and in order to determine conceptual mine dewatering and pumping requirements. Preliminary 2D FEM numerical modeling has been completed by AGP for the current study to confirm these Order of Magnitude estimates (Figure 16-19). These conceptual magnitudes will need to be reviewed once project specific hydrogeological data is collected, as required at the next level of study.

Figure 16-19: Conceptual/Order of Magnitude Inflow Estimate



16.2.5 Geotechnical Model Limitations

The preceding section summarizes information and knowledge gathered to date, primarily by others, along with information collected by AGP during a recent four-day site visit.

This information provides the basis for preliminary pit slope and underground mine designs and guidelines, to assist with PEA-level planning and cost estimating for the project.

The current geotechnical dataset is considered adequate for conceptual level designs. Where data gaps exist, the engineering geology of the area has been inferred from available data. When quantifying material properties of the rock, ranges of values have been estimated.

Engineering geology interpretations presented in this report should be considered preliminary. Data collected to date may not accurately reflect the rock mass comprising the final open pit walls or underground excavations. Where appropriate, geological features identified should be verified and validated with additional field work and interpretation.

16.2.6 Data Gap Analysis

A geotechnical data gap analysis has been completed by AGP to determine data requirements to support a PFS level mine design for the proposed open pit and underground workings. (Table 16-5)

The available data was evaluated relative to the following considerations:

- Spatial Coverage - ensuring sufficient coverage of rock mass quality and discontinuity orientations of the rock masses in the walls of each major sector of the open pit mine and HW, Ore Zone, and FW zones underground

- Geological Coverage - ensuring sufficient characterization of the different geological units (lithologies) expected in the vicinity of the workings
- Coverage of Major Features - ensuring known faults and other features have been intersected and characterized
- Orientation Data Bias - ensuring the discontinuity orientation data is sufficiently free of directional bias
- Orientation Data Quality - ensuring the discontinuity orientation data is of suitable quality
- Laboratory Strength Testing - ensuring sufficient laboratory strength testing has been completed to characterize the intact rock properties of the different geological units expected

Table 16-5: Geotechnical Data Gap Analysis

Gap Analysis Criteria	Status	Gaps
Spatial Coverage	Pit: Good UG : Fair	Limited detailed geotechnical data has been collected to date within the pit and underground mining zones; drill holes do not intersect large portions of the proposed expanded pit walls or underground mine areas Detailed geotechnical data, and structural data, is required for all slope sectors
Geological Coverage	Pit: Good UG : Fair	Moderate geological data and limited geotechnical data (mainly from past mining) currently exists for the project area Limited spatial and geotechnical knowledge exists regarding location and intensity of fault impacted zones
Coverage of Major Features	Pit: Good UG : Fair	Structural characterization work completed by SRK (2018), SRK identified additional work requirements Limited orientation and persistence data available
Orientation Data Bias	Pit: Good UG : Limited	Orientation data and analysis required (evaluated using coring data and / or ATV/OTV surveys)
Orientation Data Quality	Pit: Good UG : Limited	Orientation data and analysis required (evaluated using coring data and / or ATV/OTV surveys)
Field and Laboratory Strength Testing	Limited	Limited rock strength estimates, mainly from past open pit mining studies UCS, tri-axial, tensile, direct shear and other standard laboratory tests are required to determine / confirm rock strength & deformation parameters, discontinuity strength criteria
Hydrogeology	Pit: Good UG : Limited	Conceptual PEA inflow estimates determined from first principles and basic modeling will need to be investigated and updated at the next level of study

The results of the geotechnical gap analysis indicate several important factors that require additional investigation. For PFS designs, a higher level of confidence is required, and the preliminary geotechnical model presented will need to be updated with additional higher-quality data, A series of additional recommendations on data collection and interpretation tasks are outlined in Section 26 of this report.

16.3 Open Pit

16.3.1 Geologic Model Importation

The 2020 resource estimates were created using GEMS software for mineralization domains and block modelling. Troilus Gold provided AGP with resource models in CSV block model format for open pit and underground mine planning. The final resource model provided to AGP for Z87 and J (J4/J5) mine design was a whole block model while the SW resource model included an ore percentage item.

Framework details of the different open pit block models are provided in Table 16-6. The final mine planning model items for J and Z87 are displayed in Table 16-7 and for SW in Table 16-8. The mining model created by AGP in Hexagon MinePlan® includes additional items for mine planning purposes. MinePlan® was used for the mining portion of the PEA, utilizing their Lerchs Grossman (LG) shell generation, pit and dump design and mine scheduling tools.

Table 16-6: Open Pit Model Framework

Framework Description	SW open pit model Value	J/Z87 open pit model Value
MinePlan® file 10 (control file)	sw10.dat	trl10.dat
MinePlan® file 15 (model file)	sw15.min	trl15.mp2
X origin (m)	9,100	9,000
Y origin (m)	8,700	12,200
Z origin (m) (max)	5420	5455
Rotation (degrees clockwise)	0	0
Number of blocks in X direction	95	380
Number of blocks in Y direction	165	730
Number of blocks in Z direction	53	191
X block size (m)	10	5
Y block size (m)	10	5
Z block size (m)	10	5

Only Indicated and Inferred resources were used for the PEA. No Measured resources were reported in the model provided. The block density values provided in the resource models were estimated by the geology team while the blocks without values received default values of 2.77 t/m3 in the J/Z87 model and 2.80 t/m3 in the SW model.

Table 16-7: J and Z87 Model Item Descriptions

Field Name	Min	Max	Precision	Units	Comments
TOPO	0	100	0.01	%	Topography percent (coded with July 2019 drone survey/regional hybrid)
DEN	0	5	0.01	t/cu m	Density. (J4 ore=2.77, J5 ore=2.80, Z87 ore=2.86, waste=2.77, OB=2.20)
AUEQ	0	10	0.001	g/t	Equivalent Gold (from resource model)
AU	0	10	0.001	g/t	Gold
CU	0	10	0.001	%	Copper
AG	0	10	0.001	g/t	Silver
RCLS	0	9	1	-	Resource Class (1=mea, 2=ind, 3=inf, default=9)
ZONE	0	10	1	-	Zone (1=Z87, 2=87S, 4=J4, 5=J5, 6=default)
ROCK	0	2000	1	-	Rock Type (87-87S = 11-25,1001-1003, J4-J5 = 40-55, default=0)
OVB	0	100	0.01	%	Percent below overburden surface
RTYPE	1	10	1	-	Rocktype code (1= overburden, 2=bedrock)
UGRSC	0	1	1	-	Blocks used by RPA for Jan 2019 u/g resource (0=exclude, 1=include)
UGCLS	0	9	1	-	Classification used by RPA for Jan 2019 u/g resource
VLT1	0	999	0.01	$/t	Preliminary pit shell value per tonne, Z87 pit, restricted pit bottoms
VLT2	0	999	0.01	$/t	Preliminary pit shell value per tonne, Z87 pit
VLT3	0	999	0.01	$/t	Preliminary pit shell value per tonne, J pit
VLT4	0	999	0.01	$/t	Heap pit shell value per tonne, J pit
VLT5	0	999	0.01	$/t	Heap pit shell value per tonne, Z87 pit
VLT6	0	999	0.01	$/t	Heap pit shell value per tonne, J pit - May 13 parameters and prices
NSRH7	0	999	0.01	$/t	NSR for heap leach
NSRM7	0	999	0.01	$/t	NSR for milling
VLT7	0	999	0.01	$/t	Heap/mill pit shell value per tonne, J pit - May 13 parameters and prices
DEST7	0	9	1	-	Material destination for J pit - May 13 parameters (1=heap,2=mill, 3=waste)
NSR8	0	999	0.01	$/t	NSR for con-dore split in J and 87 pits at 1550Au, 3.25Cu, 20Ag (May 29)
VLT8	0	999	0.01	$/t	RSC value per tonne for J pit using NSR8
VLT9	0	999	0.01	$/t	RSC value per tonne for J pit using RPA Dec 2019 parameters and 1550Au
NSR10	0	999	0.01	$/t	NSR for con-dore split in J and 87 pits at 1375Au, 2.80Cu, 17Ag (May 30), PEA base case
VLT10	0	999	0.01	$/t	Value per tonne for J pit using NSR10
VLT11	0	999	0.01	$/t	Value per tonne for J pit using NSR10 (from RF0.5-1 pit shells)
WALLJ	1	2	1	-	J pit wall code (1=east, 2=N,S,W)
BERMJ	0	99	0.01	m	J pit berm widths for pit design
VLT12	0	999	0.01	$/t	Value per tonne for 87 pit using NSR10
VLT13	0	999	0.01	$/t	RSC value per tonne for J pit using NSR8 (RPA slopes used)
VLT14	0	999	0.01	$/t	RSC value per tonne for Z87 pit using NSR8
VLT15	0	999	0.01	$/t	RSC value per tonne for Z87 pit using RPA Dec 2019 parameters and 1550Au
VLT16	0	999	0.01	$/t	RSC value per tonne for Z87 pit using NSR8 (RPA slopes used)
NSR11	0	999	0.01	$/t	NSR/t with elevated Au price (1600Au, 20Ag, 3.25Cu)
NSR12	0	999	0.01	$/t	NSR/t with elevated Au price (1650Au, 20Ag, 3.25Cu)
NSR13	0	999	0.01	$/t	NSR/t with elevated Au price (1700Au, 20Ag, 3.25Cu)
VLT17	0	999	0.01	$/t	$/t of mineralization with NSR11 (1600Au, 20Ag, 3.25Cu) in J pit
VLT18	0	999	0.01	$/t	$/t of mineralization with NSR12 (1650Au, 20Ag, 3.25Cu) in J pit
VLT19	0	999	0.01	$/t	$/t of mineralization with NSR13 (1700Au, 20Ag, 3.25Cu) in J pit
VLT20	0	999	0.01	$/t	$/t of mineralization with NSR11 (1600Au, 20Ag, 3.25Cu) in Z87 pit

VLT21	0	999	0.01	$/t	$/t of mineralization with NSR12 (1650Au, 20Ag, 3.25Cu) in Z87 pit
VLT22	0	999	0.01	$/t	$/t of mineralization with NSR13 (1700Au, 20Ag, 3.25Cu) in Z87 pit
WALLZ	1	2	1	-	87 pit wall code (1=east, 2=N,S,W)
BERMZ	0	99	0.01	m	87 pit berm widths for pit design
DILBO	0	4	1	-	count of mill feed blocks touching each waste block
IFLAG	0	1	1	-	In-situ block flag where 1=ore, 2=waste
DORE%	0	100	0.01	%	final percent of block to include as diluted ore
DWAS%	0	100	0.01	%	final percent of block to include as waste
MINE	0	1	1	-	entire model coded as 1. Used in dilution calculations.
VLT23	0	999	0.01	$/t	$/t of mineralization with NSR11 (1600Au, 20Ag, 3.25Cu) in J pit, July 20 PC and G&A
VLT24	0	999	0.01	$/t	$/t of mineralization with NSR11 (1600Au, 20Ag, 3.25Cu) in Z87 pit, July 20 PC and G&A
DAUEQ	0	99	0.001	g/t	diluted gold equivalent grade (using base case parameters)

Table 16-8: SW Model Item Descriptions

Field Name	Min	Max	Precision	Units	Comments
TOPO	0	100	0.01	%	Topography percent (coded with regional topo from RSC model)
DEN	0	5	0.01	t/cu m	Density (variable by ROCK code, waste=2.80, OB=2.20)
ORE%	0	100	0.001	%	Percent Ore
AUEQ	0	10	0.001	g/t	Equivalent Gold (from resource model)
AU	0	10	0.001	g/t	Gold
CU	0	10	0.001	%	Copper
AG	0	15	0.001	g/t	Silver
AUR	0	10	0.001	g/t	Gold recovered
CUR	0	10	0.001	%	Copper recovered
AGR	0	15	0.001	g/t	Silver recovered
RCLS	0	9	1	-	Resource Class (1=mea, 2=ind, 3=inf, default=9)
ZONE	0	10	1	-	Zone (3=SW, 6=default)
ROCK	0	2000	1	-	Rock Type (201-208, default=0)
OVB	0	100	0.01	%	Percent below overburden surface
RTYPE	1	10	1	-	Rocktype code (1= overburden, 2=bedrock)
VLT1	0	999	0.01	$/t	Preliminary mineralized value per tonne, excluding mining costs
VLT2	0	999	0.01	$/t	Preliminary RSC value per tonne, excluding mining costs
VLT3	0	999	0.01	$/t	Preliminary RPA RSC value per tonne, excluding mining costs (in USD)
VLT4	0	999	0.01	$/t	Preliminary RSC value per tonne, excluding mining costs (RPA slopes)
NSR5	0	999	0.01	$/t	NSR/t using May 23 base case parameters
VLT5	0	999	0.01	$/t	$/t of mineralization using May 23 base case parameters, excluding mining
VLB5	-9999	999999	1	$	Block value using May 23 base case parameters
NSR6	0	999	0.01	$/t	NSR/t using May 26 base case parameters
NSR6D	0	999	0.01	$/t	NSR/t using May 26 base case parameters after dilution, use diluted grades and
VLT6	0	999	0.01	$/t	$/t of mineralization using May 26 base case parameters, excluding mining
VLB6	-9999	999999	1	$	Block value using May 26 base case parameters
SLOPE	0	9	1	-	slope zone for pit design (1=default, 2=east wall)
NSR7	0	999	0.01	$/t	NSR/t using May 26 RSC parameters (1550Au, 20Ag, 3.25Cu)
VLT7	0	999	0.01	$/t	$/t of mineralization using May 26 RSC parameters, excluding mining
VLT8	0	999	0.01	$/t	$/t of mineralization using 2019 Dec RPA parameters (except 1550 Au),
VLT9	0	999	0.01	$/t	$/t of mineralization using May 26 RSC parameters, excluding mining (run with
NSR8	0	999	0.01	$/t	NSR/t with elevated Au price (1600Au, 20Ag, 3.25Cu)
NSR9	0	999	0.01	$/t	NSR/t with elevated Au price (1650Au, 20Ag, 3.25Cu)
NSR10	0	999	0.01	$/t	NSR/t with elevated Au price (1700Au, 20Ag, 3.25Cu)
VLT10	0	999	0.01	$/t	$/t of mineralization with NSR8 (1600Au, 20Ag, 3.25Cu)
VLT11	0	999	0.01	$/t	$/t of mineralization with NSR9 (1650Au, 20Ag, 3.25Cu)
VLT12	0	999	0.01	$/t	$/t of mineralization with NSR10 (1700Au, 20Ag, 3.25Cu)
FLAG	0	5	1	-	Dilution flag: 1=mineralized ore, 2=mineralized waste, 3=waste
DILBK	0	4	1	-	number of waste blocks touching an mill feed block
DILBO	0	4	1	-	number of mill feed blocks touching a waste block
DORE%	0	100	0.01	%	diluted mill feed %
DWAS%	0	100	0.01	%	diluted waste %
DAU	0	10	0.001	g/t	diluted gold grade
DAG	0	15	0.001	g/t	diluted silver grade
DCU	0	10	0.001	%	diluted copper grade
MINE	0	1	1	-	entire model coded as 1. Used in dilution calculations.
VLT47	0	999	0.01		$/t of mineralization using July 20 RSC parameters, excluding mining (run with
DAUEQ	0	99	0.001	g/t	diluted gold equivalent grade (using base case parameters)

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 preliminary assessments by AGP (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 16-9.

Table 16-9: Pit Shell Slopes

Pit	Azimuth (degrees)	Material Type	Overall Slope (degrees)	Description
J/Z87	**0-360**	**Overburden**	**30**	
	30	Bedrock	49.2	One 33.2m wide ramp, IRA = 53 deg, wall height of 300m
	45	Bedrock	43.8	One 33.2m wide ramp, IRA = 47 deg, wall height of 300m
	135	Bedrock	43.8	One 33.2m wide ramp, IRA = 47 deg, wall height of 300m
	150	Bedrock	49.2	One 33.2m wide ramp, IRA = 53 deg, wall height of 300m
SW	**0-360**	**Overburden**	**30**	
	30	Bedrock	48.3	One 33.2m wide ramp, IRA = 53 deg, wall height of 240m
	45	Bedrock	43	One 33.2m wide ramp, IRA = 47 deg, wall height of 240m
	135	Bedrock	43	One 33.2m wide ramp, IRA = 47 deg, wall height of 240m
	150	Bedrock	48.3	One 33.2m wide ramp, IRA = 53 deg, wall height of 240m

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 APG in consultation with Troilus personnel.

The parameters used are shown in Table 16-10. The values are in United States dollars unless otherwise noted. Costs and revenues are converted to Canadian dollars for use in pit shell determination. The mining cost estimates are based on the use of 181 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 and SW pits in this study were 12%, 23% and 18% respectively.

Table 16-10: Economic Pit Shell Parameters (US Dollars unless otherwise noted)

Description			Units	Value		
Exchange rates						
	CAD		US$ =	1.30		
Resource Model						
	Block classification used			M+I+I		
	Block Model Height			5 (J/87) and 10 (SW)		
	Mining Bench Height			10		
Metal Prices				**Copper**	**Gold**	**Silver**
	Price		$/oz	2.80	1375.00	17.00
	Royalty		%	3.5%	3.5%	3.5%
Dore						
	Payable		%		99%	
	Selling Cost		$/oz		12.00	
Smelting, Refining, Transportation Terms						
	Payable		%	variable	96.5%	90.0%
	Minimum Deduction		unit, g/dmt	variable	0	30
	Participation (on profits)		%	100%	100%	100%
	Smelting Charge		$/dmt	62.00		
	Refining		$/oz, $/lb	0.062	5.00	0.50
	Concentrate Moisture		%	8%		
	Transit Losses		%	0.5%	0	0
	Concentrate Trucking Cost		$/wmt	70.30		
Metallurgical Information						
	J Zone			**Copper**	**Gold**	**Silver**
		Recovery - Global	%	90%	90%	40%
		Recovery - Gravity	%		30%	
		Recovery - Flotation	%	90%	60%	40%
		Concentrate Grades	% or g/t	12%		
	87 Zone					
		Recovery-Global	%	90%	90%	40%
		Recovery - Gravity	%		30%	
		Recovery - Cyanidation	%		0%	0%
		Recovery - Flotation	%	90%	60%	40%
		Concentrate Grades	% or g/t	23%		
	SW Zone					
		HG Recovery-Global	%	92%	90%	40%
		Recovery - Gravity	%		30%	
		Recovery - Flotation	%	92%	60%	40%
		LG Recovery-Global	%	90%	88%	40%
		Recovery - Gravity	%		30%	
		Recovery - Flotation	%	90%	58%	40%
		Concentrate Grades	% or g/t	18%		
Power Cost						
	Cost of power		C$/Kwhr	0.033		

Fuel Cost					
	Diesel Fuel Cost to site	C$/ l	1.06		
Mining Cost *					
	Base Rate - 5390 Elevation (5370 for SW)		**J Zone**	**87 Zone**	**SW**
	Waste	C$/t moved	1.68	1.68	1.53
	Mill Feed	C$/t moved	1.82	1.82	1.67
	Incremental Rate - below 5390 elevation (5370 for SW)				
	Waste	C$/t moved/ 10m bench	0.029	0.029	0.034
	Mill Feed	C$/t moved/ 10m bench	0.023	0.023	0.013
Processing and G&A					
	Processing Cost	C$/t mill feed	6.80		
	G&A Cost	C$/t mill feed	2.36		
	Process + G&A	C$/t mill feed	9.16		

* mining costs based on using 181 t haul trucks
* preliminary process costs based on 20,000 tpd

Nested LG pit shells were generated to examine sensitivity to metal prices with base case prices of US$1,375/oz Au, US$2.80/lb Cu and US$17.00/oz Ag. This was done to gain an understanding of the deposit and highlight potential opportunities in the design process to follow. Undiluted Indicated and Inferred material was used in the 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$188/oz up to US$1,650/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 J pit profit versus price is displayed in Figure 16-20. Using this initial graph, it was found to be difficult to select individual phases since there are significant changes over a narrow price range. For this reason, another set of pit shells was calculated using a tighter price range as displayed in Figure 16-21.

Figure 16-20: J Pit Profit vs. Price by Pit Shell

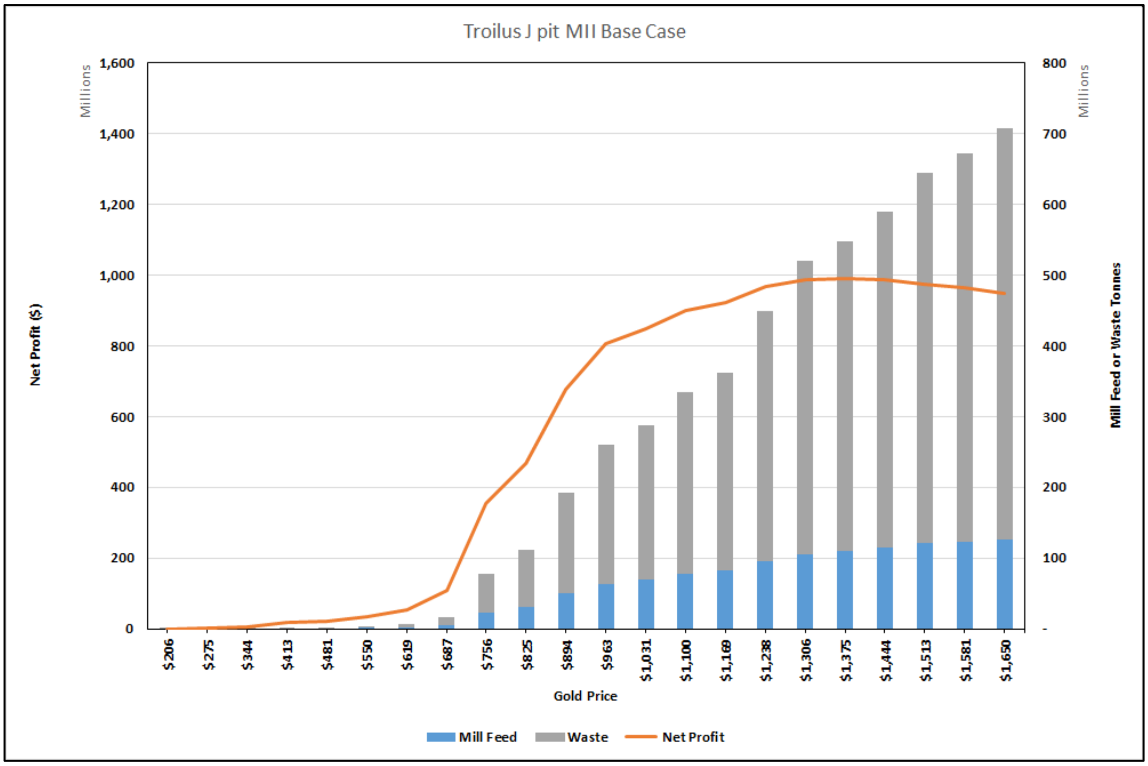


Figure 16-21: J Pit Profit vs. Price by Pit Shell (more detailed)

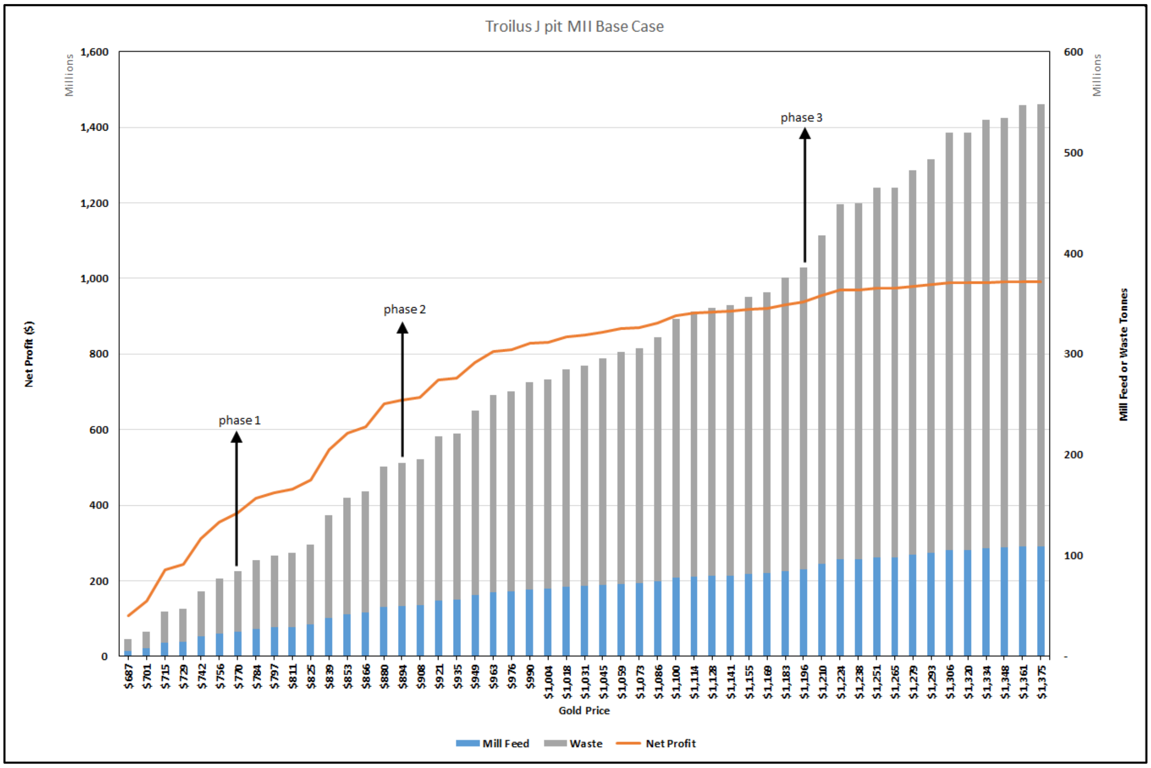


Figure 16-22 illustrates various break points in the pit shells. With each incremental increase in the waste tonnage, and to a lesser degree the mill tonnage, the undiscounted net profit also increased. In the case of the first break point shown at US$770/oz Au, the cumulative waste tonnage is 60.0 Mt, with a corresponding mill feed tonnage of 24.4Mt or a strip ratio of 2.5:1. 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 38% of the net value of a $1,375/oz pit but with only 14% of the waste of the larger pit shell. This pit shell was used for the pit design of Phase 1.

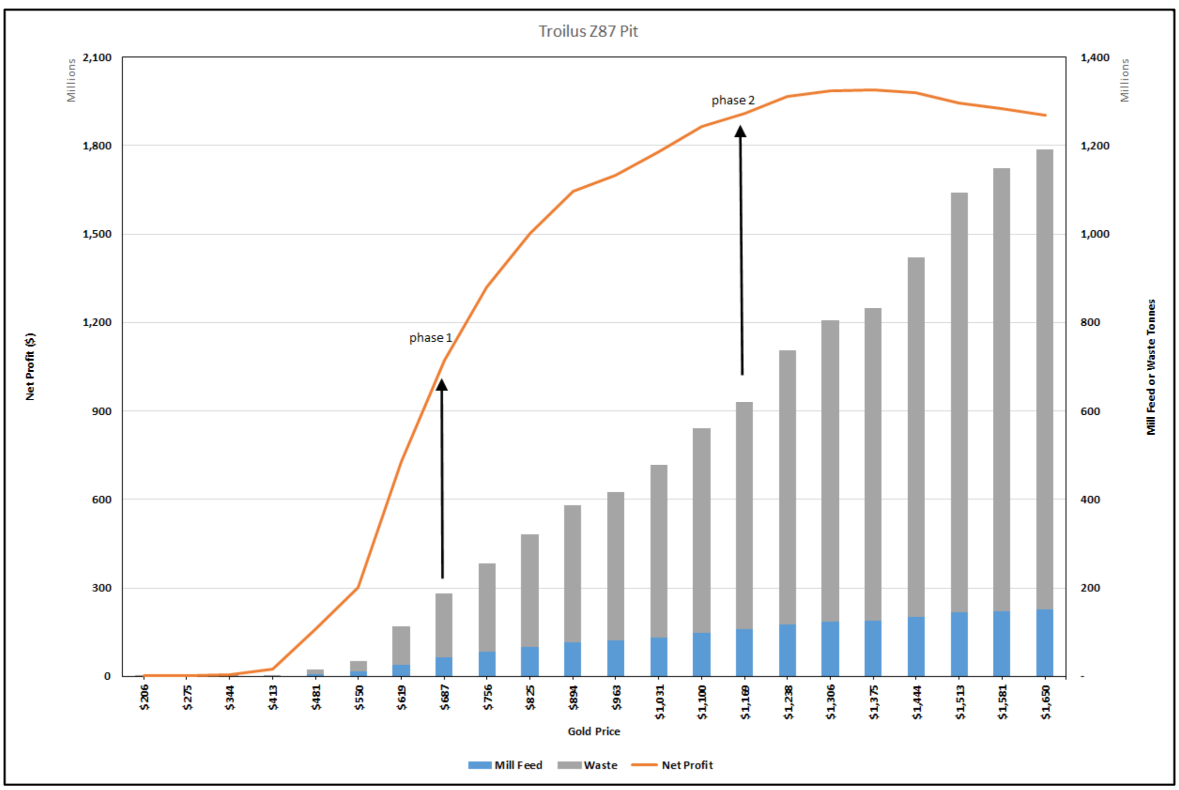
The second break point was at US$894/oz Au. The incremental waste tonnage from the first break point is 82.1 Mt, with a corresponding increase in mill feed tonnage of 25.7 Mt or a strip ratio of 3.2:1. The net profit also increased beyond this point showing that there was still value to be obtained by going with a higher metal price. This pit shell was used for the pit design of Phase 2. There is a significant waste tonnage in the next higher pit price to achieve the next increase in profit. The cumulative value of the first two selected pit shells was 68% of the US$1,375/oz Au pit shell but with only 32% 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 third major break point was at US$1196/oz Au. This resulted in a substantial jump in the waste tonnage from the second break point by 157.6 Mt with a gain of 36.5 Mt of feed material for an incremental strip ratio of 4.3:1. The cumulative value of the first three break points was 95% of the US$1,375/oz Au pit shell but with only 68% of the waste movement of the larger pit required. 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.

The plot of Z87 pit profit versus price is displayed in Figure 16-22 and illustrates various break points in the pit shells. In the case of the first break point shown at US$687/oz Au, the cumulative waste tonnage is 144.1 Mt, with a corresponding mill feed tonnage of 43.4Mt or a strip ratio of 3.3:1. 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 54% of the net value of a $1,375/oz pit but with only 20% of the waste of the larger pit shell. This pit shell was used for the pit design of Phase 1.

The second break point was at US$1169/oz Au. The incremental waste tonnage from the first break point is 369.5 Mt, with a corresponding increase in mill feed tonnage of 62.3 Mt or a strip ratio of 5.9:1. This pit shell was used for the pit design of Phase 2. There is significant waste tonnage in this increment and would need to be evaluated in a schedule to see whether or not it is better than going underground below phase 1. The cumulative value of the first two selected pit shells was 96% of the US$1,375/oz Au pit shell but with only 73% 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.

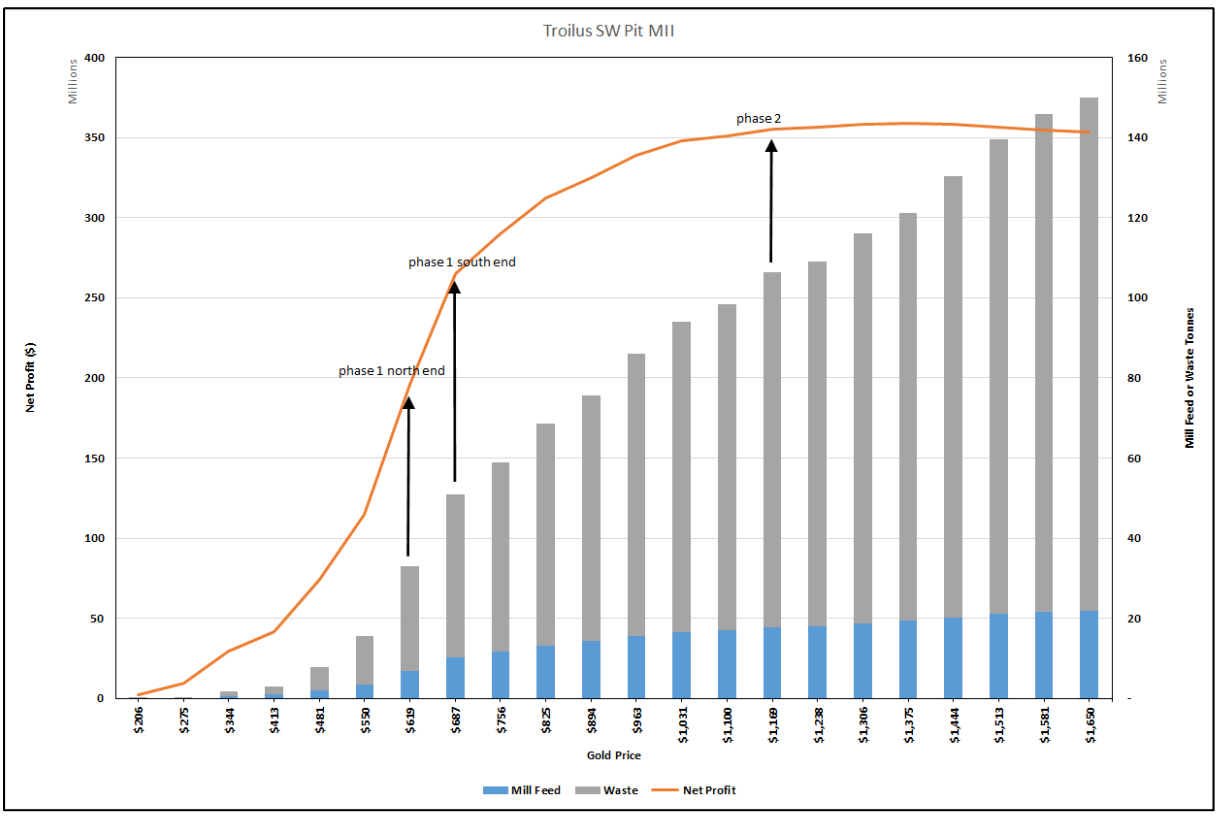
Figure 16-22: Z87 Pit Profit vs. Price by Pit Shell



The plot of SW pit profit versus price is displayed in Figure 16-23 and illustrates various break points in the pit shells. In the case of the first break point shown at US$619/oz Au, the cumulative waste tonnage is 26.1 Mt, with a corresponding mill feed tonnage of 7.0 Mt or a strip ratio of 3.8:1. 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 55% of the net value of a $1,375/oz pit but with only 26% of the waste of the larger pit shell. A second break point was identified at US$687/oz, with cumulative tonnage of 40.6 Mt and a corresponding cumulative mill feed tonnage of 10.2 Mt or a strip ratio of 3.8:1. The US$619/oz Au pit shell appeared appropriate for the north end of the phase 1 design where access would be difficult at depth within the ultimate pit. The US$687/oz Au pit shell similarly appeared to be more appropriate for the south end of the phase 1 design where shallower access would be required within the ultimate pit.

The next significant break point was at US$1169/oz Au. The incremental waste tonnage from the first break point is 62.3 Mt, with a corresponding increase in mill feed tonnage of 10.8 Mt or a strip ratio of 5.7:1. This pit shell was used for the pit design of Phase 2. The cumulative value of the final selected pit shell was 99% of the US$1,375/oz Au pit shell but with only 87% 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 16-23: SW Pit Profit vs. Price by Pit Shell



16.3.3 Dilution

Two different open pit resource model types were provided for the Troilus deposits. The J/Z87 model is a whole block format while the SW model is an ore percentage format. Different approaches were used to apply external dilution in an appropriate manner for each model. The different dilution approaches are described in the following sub-sections.

SW Model Dilution

The SW undiluted resource model is an ore percentage format. This means the grades from the wireframes were reported into separate percentage parcels of mill feed and waste in each block.

To account for mining dilution, AGP modeled contact dilution into the in-situ resource blocks. To determine the amount of dilution, and the grade of the dilution, the size of the block in the model was examined. The block size within the model was 10 x 10 m in plan view, and 10 m high. Mining would be completed on 10 m benches for mill feed and waste. Five metre lifts for mill feed will be implemented if required and the equipment selected is capable of mining in that manner.

The percentage of dilution is calculated for each contact side using an assumed 1.0 m contact dilution distance. This dilution skin thickness was selected by considering the spatial nature of the mineralization, proposed grade control methods, GPS assisted digging accuracy, and blast heave.

If one side of a mineralized block above cut-off is in contact with a waste block, then it is estimated that dilution of 10% (1m / 10m) would result. If two sides are contacting, it would rise to 20%. Three sides would be 30%, and four sides 40%. Four sides represent an isolated block of mill feed.

Not all mineralized blocks (ORE% > 0) in the resource model contain grade values, however the material outside the mineralized shapes have no grade estimates and have been treated as though the Au and Ag grades are zero for dilution purposes. The net value per tonne that was stored to the block model during the LG runs was used as the grade for cut-off application. As that net value per tonne is inclusive of all on-site operation costs except for mining, applying a $0.01/t cut-off represents the marginal cut-off grade to flag initial feed and waste blocks. Using this marginal cut-off grade, all model blocks were flagged as either (1) feed blocks, (2) waste blocks within mineralized material, or (3) default waste blocks outside of mineralization.

MinePlan® has a routine called gndiln.dat that enables the user to query surrounding blocks against a set of conditions. AGP applied a two-pass approach to all model block dilution calculations in order to define new items called DORE%, DWAS%, DAU, DCU and DAG. The default waste blocks would receive DORE%=0 and diluted gold and silver grades of 0 g/t as well as a diluted copper grade of 0%. The two-pass dilution calculations are summarized as follows:

- For the first pass of dilution calculations, the procedure was run to determine how many waste blocks contacted a feed block. A dilution percentage with no grade was then added to the original ore percentage to determine a new diluted ore percentage up to a maximum of either 100% or the TOPO item. The assumption was that the feed block was at the edge of the mineralization when 0% < ORE% < 100%, so it could be diluted with barren waste material.

- For the second pass of dilution calculations, the procedure was run to determine how many feed blocks contacted a waste block from within the mineralization area. A dilution percentage at the waste block grade was then added as the new diluted ore percentage. The assumption was that only the feed contact sides of the mineralized waste blocks would be added as dilution.

In this manner, the contact diluted blocks were included in the tonnage and grade calculation of mill feed tonnes. The mill feed tonnage report was then run with the block model DORE% item to report out the diluted tonnes and grade.

Comparing the in-situ to the diluted values for the designed final SW pit, the diluted feed contained 14.0% more tonnes and 12.6% lower gold grade than the in-situ feed summary. The grade percentage change is lower than the feed tonnage percentage change since the mineralized waste blocks contained very low or no grades. AGP considers these dilution percentages to be reasonable considering the expected seasonal working conditions as well as the narrow mineralized zones.

J/Z87 Model Dilution

The resource model provided for the J/Z87 deposits 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 to determine the amount of dilution and the grade of the dilution. The block size within the model was 5 m by 5 m in plan, and 5 m high.

The percentage of dilution is calculated for each contact side using an assumed 0.5 m contact dilution distance. If one side of the block is touching waste, then it is estimated that dilution of 10% would result. If two sides are contacting, it would rise to 20%. Three sides would be 30%, and four sides 40%. Four sides represent an isolated block of ore.

Because the model from Troilus contained whole blocks already, the percentage of dilution could be estimated and then included in the block ore percentage item. The mining model was modified to include an ore percent item, and any blocks above the marginal cut-off grade were assigned an ore percent of 100% (deemed entirely ore). The net value per tonne that was stored to the block model during the LG runs was used as the grade for cut-off application. As that net value per tonne is inclusive of all on-site operation costs except for mining, applying a $0.01/t cut-off represents the marginal cut-off grade to flag initial feed and waste blocks. Using this marginal cut-off grade, all model blocks were flagged as either mill feed blocks or waste blocks.

The MinePlan® routine gndiln.dat was again run to determine how many mill feed blocks contacted a waste block, which determined the dilution percentage to apply. This was stored in the waste block and the waste block grade used as the diluting value. If a waste block was only surrounded by other waste blocks, the dilution percentage was zero.

In this manner, the contact blocks could be included in the tonnage and grade calculation of mill feed tonnes. The mill feed tonnage was then run with the block model DORE% item to report out the proper tonnes and grade. The block grades themselves were not modified.

Comparing the in-situ to the diluted values for the designed pit phases showed an increase in mill feed tonnage along with a lowering of gold grade. For the J pit (phases 1-3), the diluted mill feed contained 6.7% more tonnes and 6.2% lower grades than the in-situ mill feed summary. For the Z87 pit (phase 1-2), the diluted mill feed contained 5.1% more tonnes and 4.9% lower grades than the in-situ mill feed summary. The relative grade change is lower than tonnage change the due to some of the waste blocks containing mineralization.

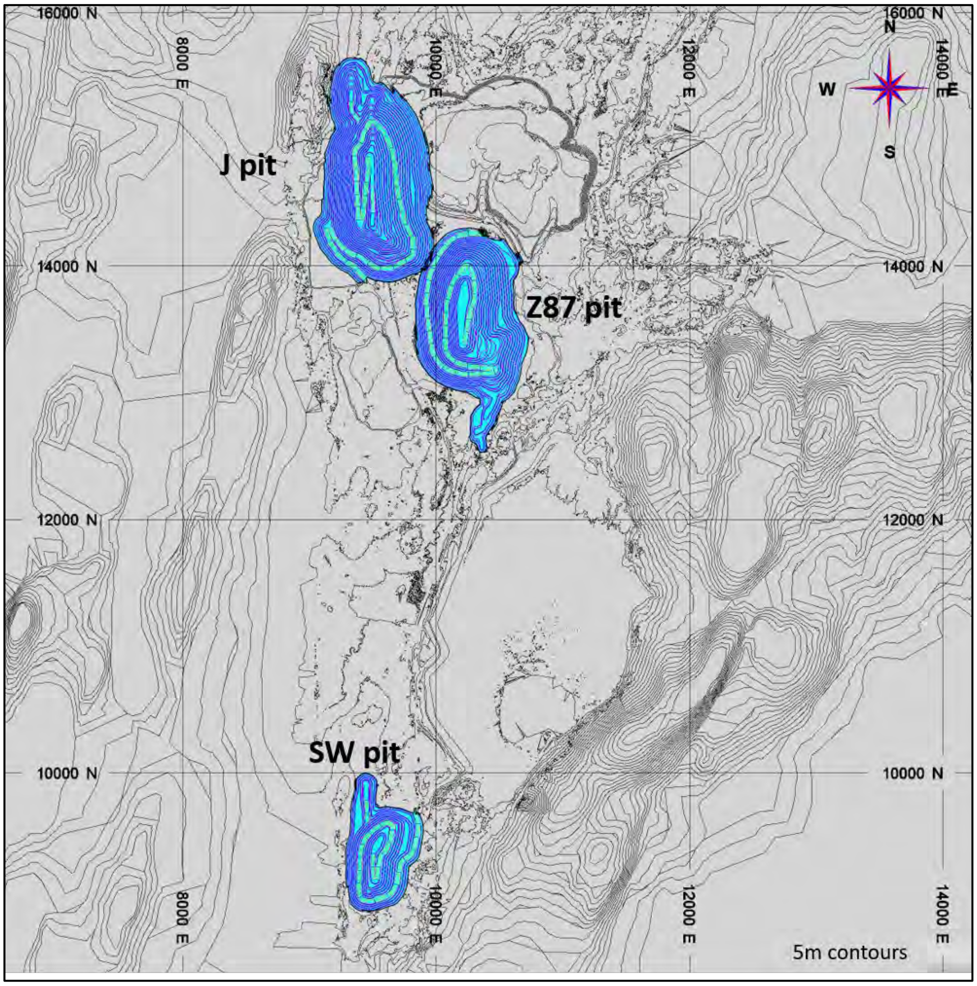
The contact dilution is lower in the J/Z87 pits than the SW pit partially due to larger thicknesses of the mill feed zones where not all the mill feed blocks are diluted with waste. The "internal dilution" from the conversion of wireframe to whole block is not included in these comparisons, otherwise the dilution tonnages would have been higher.

Tonnes and grade for the pit designs are reported with the diluted tonnes and grade.

16.3.4 Pit Design

Pit designs were developed for the J, Z87 and SW pit areas. The pit locations are displayed in Figure 16-24. The J pit design consists of three phases of successive pushbacks around the entire pit perimeter. The final Z87 pit design includes a modified phase 1 design which transitions to underground mining at 5030m elevation. The original phase 1 and 2 designs are included separately in the alternate scenario discussed in Section 24. The SW pit design consists of two small phases which can be scheduled like a satellite pit. 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.

Figure 16-24: Base Case Pit Locations



Geotechnical parameters discussed in Section 16.2 were applied to pit designs as shown in Table 16-11. The parameters agree well with the final slopes of the previously mined J and Z87 pits.

Table 16-11: Pit Design Slope Criteria

Wall Slope	Material	Inter-Ramp (degrees)	Bench Face (degrees)	Height Between (m)	Catch Bench (m)
All	Overburden	30	55	10	10.32
East	Bedrock	47	75	20	13.29
North, South,	Bedrock	53	75	20	9.71

NOTE: 10m bench heights during mining

Equipment sizing for ramps and working benches is based on the use of 180 tonne rigid frame haul trucks. The operating width used for the truck is 7.7 m. This means that single lane access is 25.5 m (2x operating width plus berm and ditch) and double lane widths are 33.2 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 16-12 using the diluted tonnes and grade from the model and a mining recovery of 98% to account for additional mill feed losses. As briefly described in Section 16.3.4, positive marginal block values from the pit optimization run were used to delineate mill feed from waste.

Table 16-12: Pit Phase Tonnages and Grades

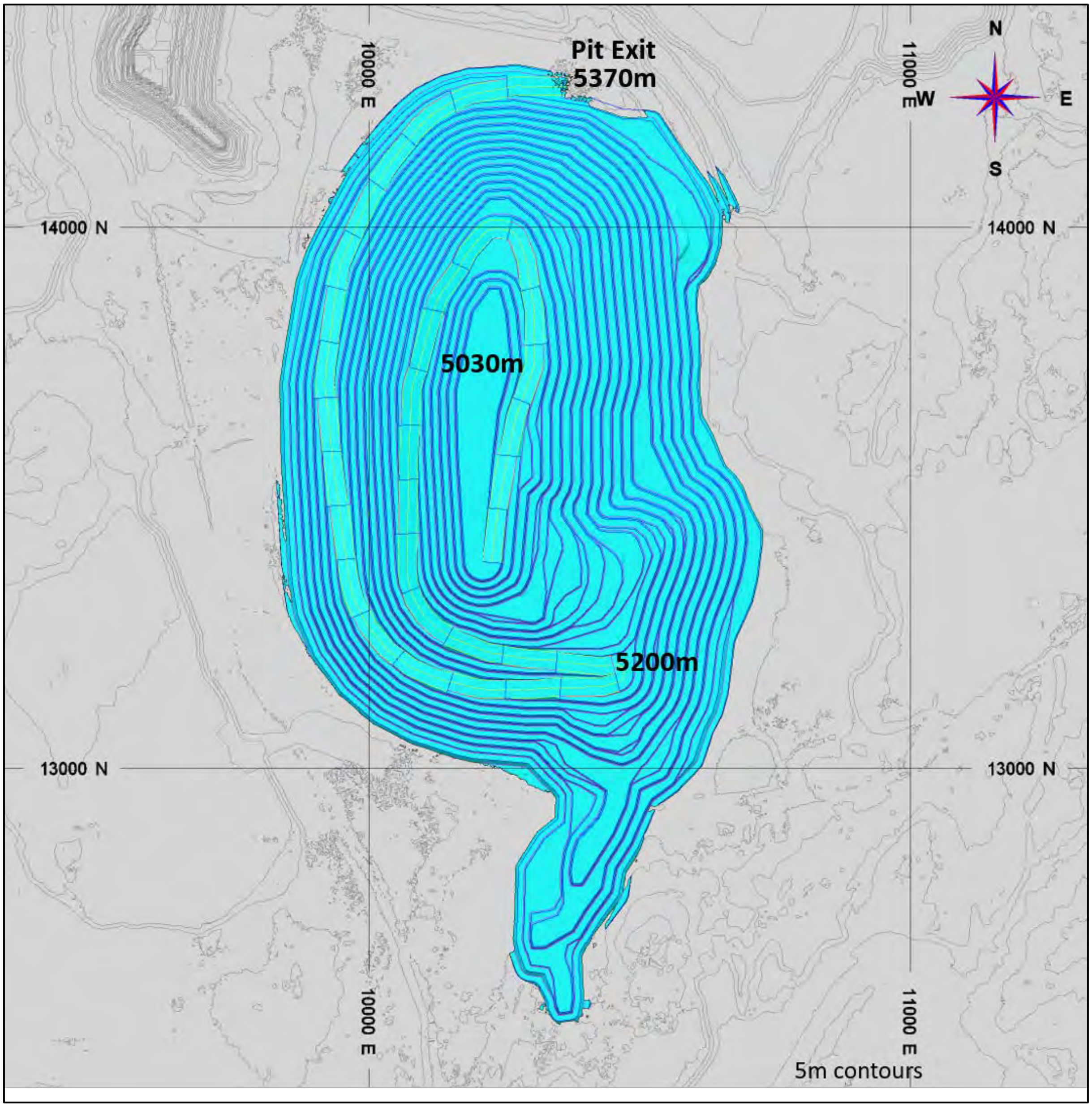
Pit	Phase	Mill Feed (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	AuEq (g/t)	Waste (Mt)	Total (Mt)	Strip Ratio
87	1	36.6	0.72	0.088	1.40	0.85	149.4	186.0	4.1
J	1	27.1	0.53	0.056	0.92	0.62	69.4	96.5	2.6
	2	28.4	0.51	0.066	0.90	0.61	103.8	132.2	3.7
	3	39.3	0.50	0.059	0.86	0.58	175.4	214.7	4.5
	Subtotal	**94.8**	**0.51**	**0.06**	**0.89**	**0.60**	**348.7**	**443.4**	**3.7**
SW	1	9.0	0.64	0.068	0.66	0.73	32.8	41.8	3.6
	2	9.8	0.65	0.062	0.85	0.74	60.2	70.0	6.2
	Subtotal	**18.8**	**0.64**	**0.065**	**0.76**	**0.74**	**93.0**	**111.8**	**5.0**
	Total	**150.2**	**0.58**	**0.068**	**1.00**	**0.68**	**591.1**	**741.2**	**3.9**

The phase designs are described in further detail in the following sections.

Z87 Phase 1

Z87 phase 1 is the first phase mined in the project. This phase includes the upper elevations of the Z87 deposit and is to be mined down to 5030 masl by open pit mining methods. Phase bench elevations range from 5390 masl down to 5030 masl. All waste and mill feed accesses will be on the north side of the pit near where the ramp exists for the historic Z87 waste dump. Shorter haul opportunities will be available in the upper benches near topography that can be utilized by the short range team. They have not been shown in the final design. A flat bottom is left on the 5030 m level to assist in the transition to underground mining at pit bottom. The design is shown in Figure 16-25.

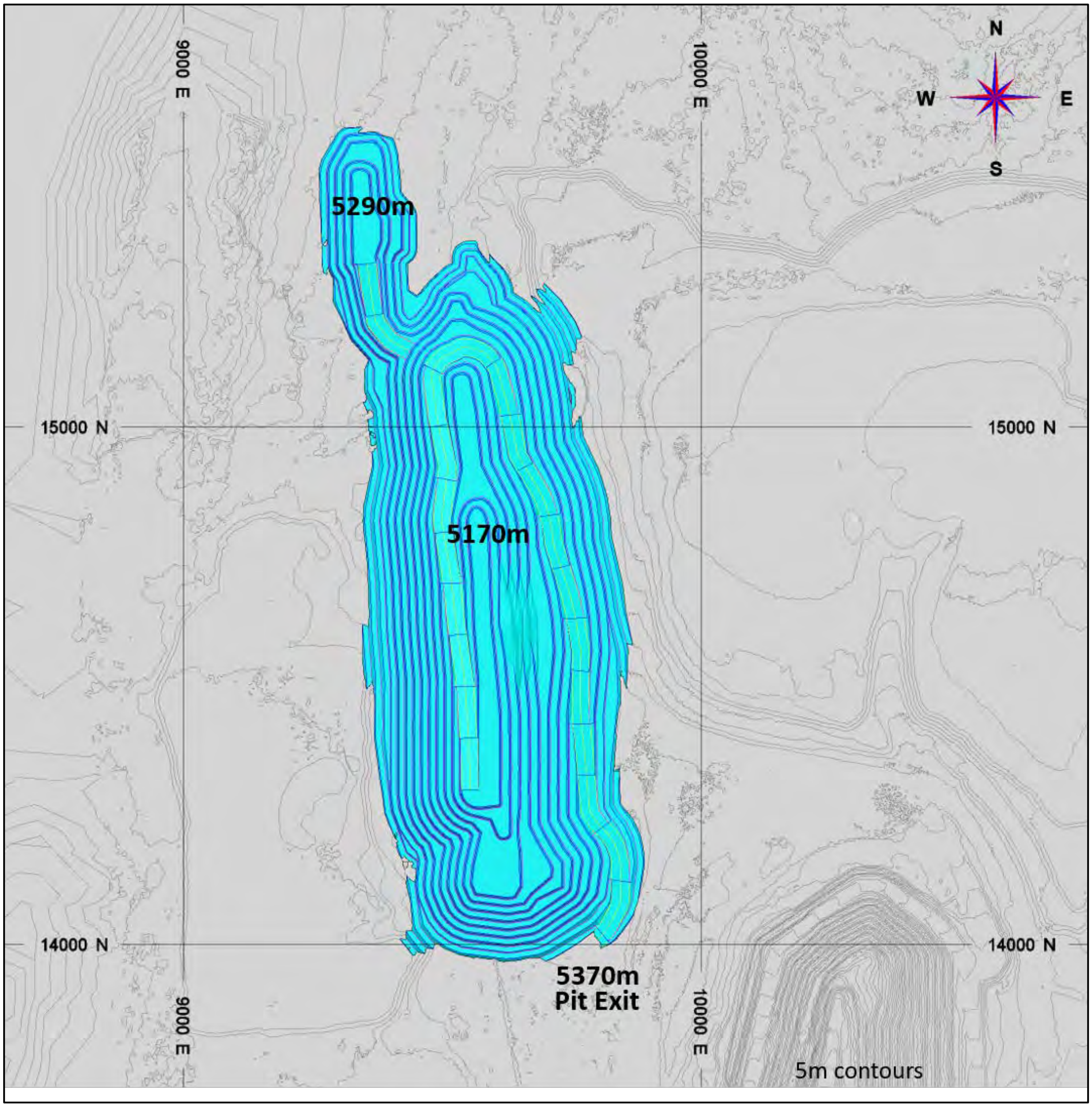
Figure 16-25: Z87 Phase 1 Design



J Phase 1

J phase 1 is the first phase mined in the J pit and it will be used to obtain early access to high value portions of the J4/J5 deposit. Phase bench elevations range from 5400 masl down to 5170 masl. The primary waste and mill feed accesses will be from the south side of the pit. Portions of the historic waste dumps to the east and west will need to be re-handled to provide stable wall slopes in J pit. The design is shown in Figure 16-26.

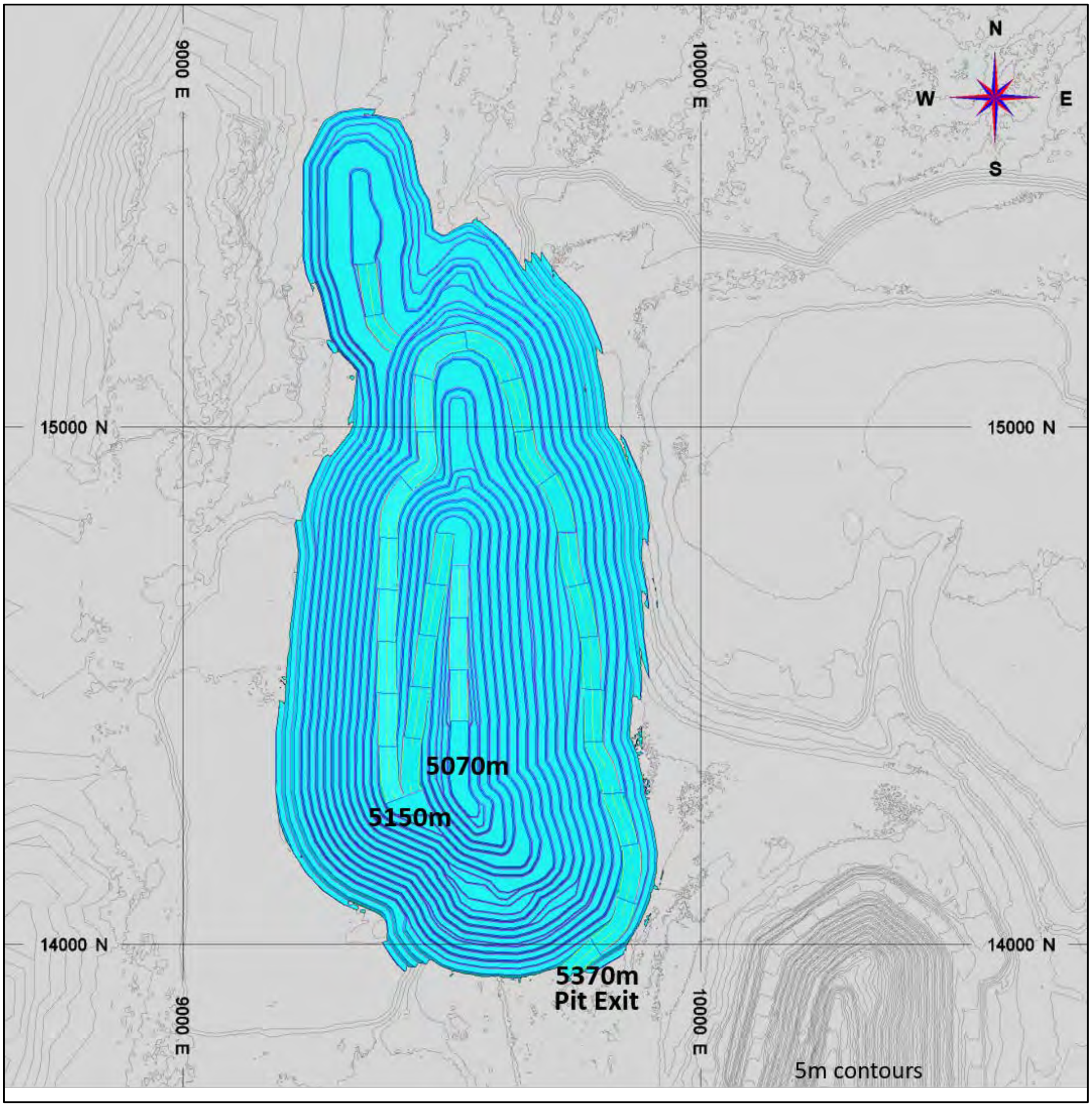
Figure 16-26: J Phase 1 Design



J Phase 2

J phase 2 is a narrow pushback after the initial phase which provides access to mill feed in the 10 benches below phase 1. Phase bench elevations range from 5400 masl down to 5070 masl. The primary waste and mill feed accesses continue to be from the south side of the pit. The design is shown in Figure 16-27.

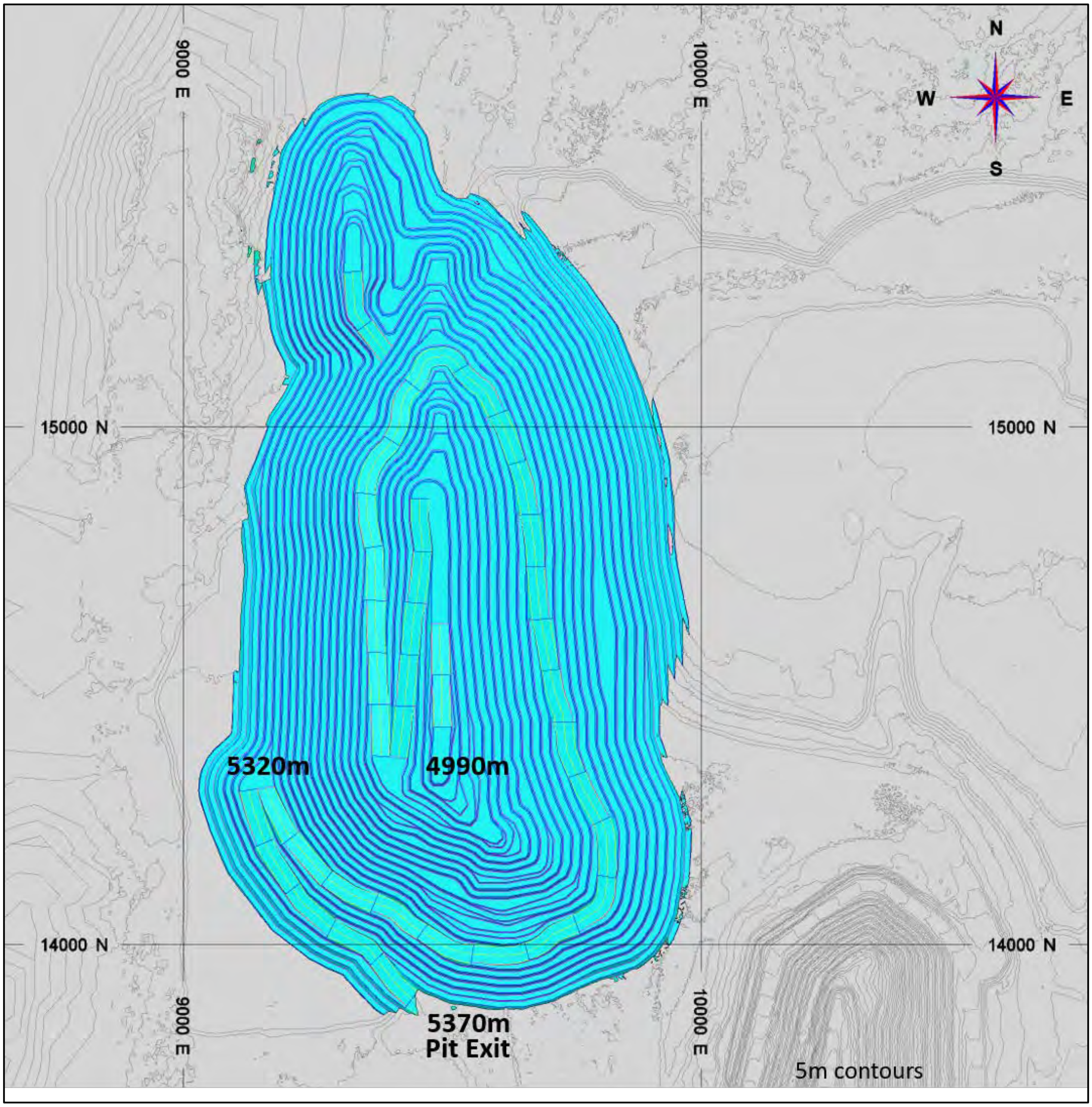
Figure 16-27: J Phase 2 Design



J Phase 3

J phase 3 is the final pushback in J pit which provides access to mill feed in the 8 benches below phase 2. Phase bench elevations range from 5410 masl down to 4990 masl. The primary waste and mill feed accesses continue to be from the south side of the pit. The design is shown in Figure 16-28.

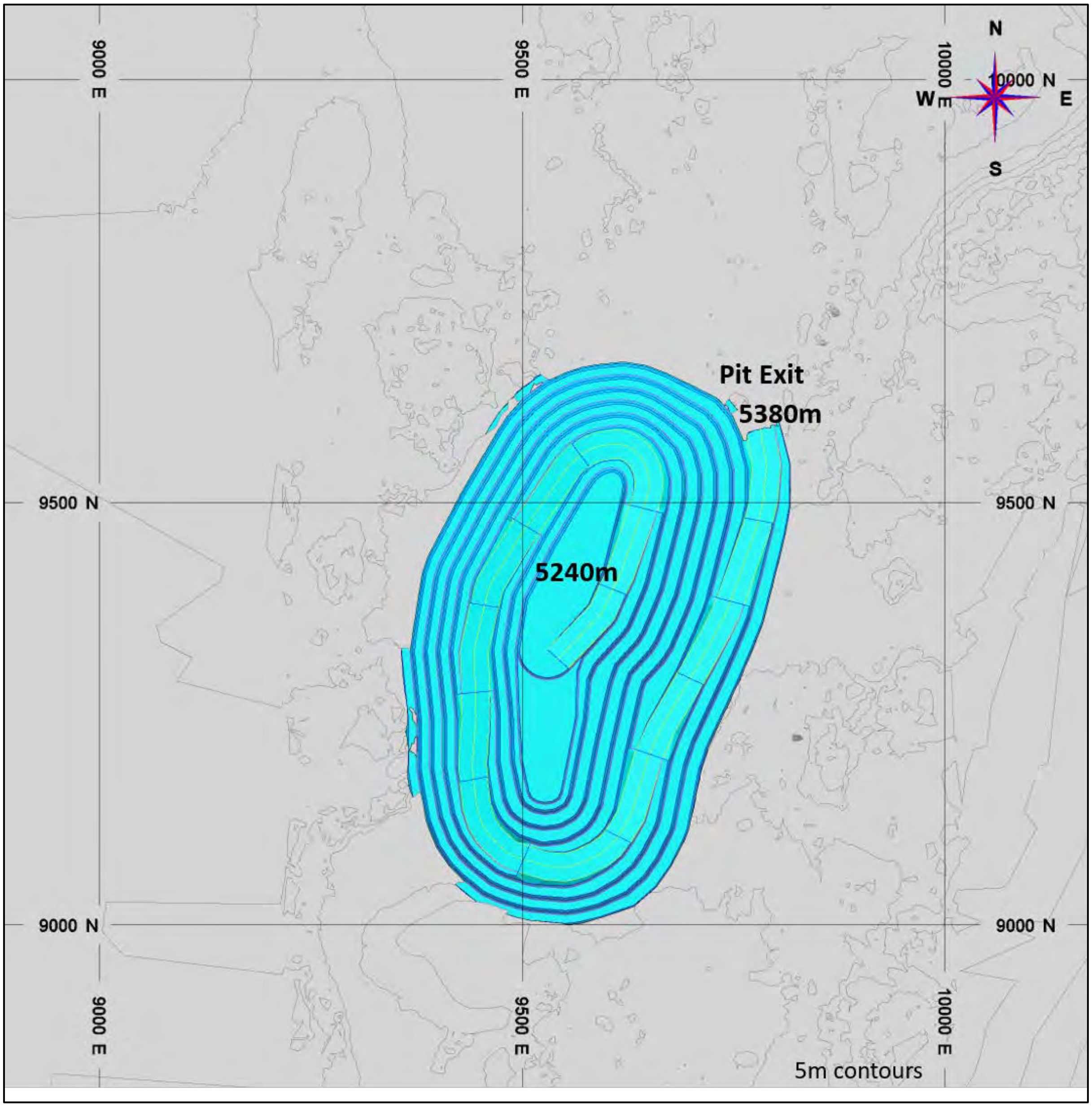
Figure 16-28: J Phase 3 Design



SW Phase 1

SW phase 1 is the first phase to be mined in this area of the project. No previous mining has occurred near the SW deposit. Phase bench elevations range from 5390 masl down to 5240 masl. All waste and mill feed accesses will be on the north side of the pit which works well for the mill location to the north. The design is shown in Figure 16-29.

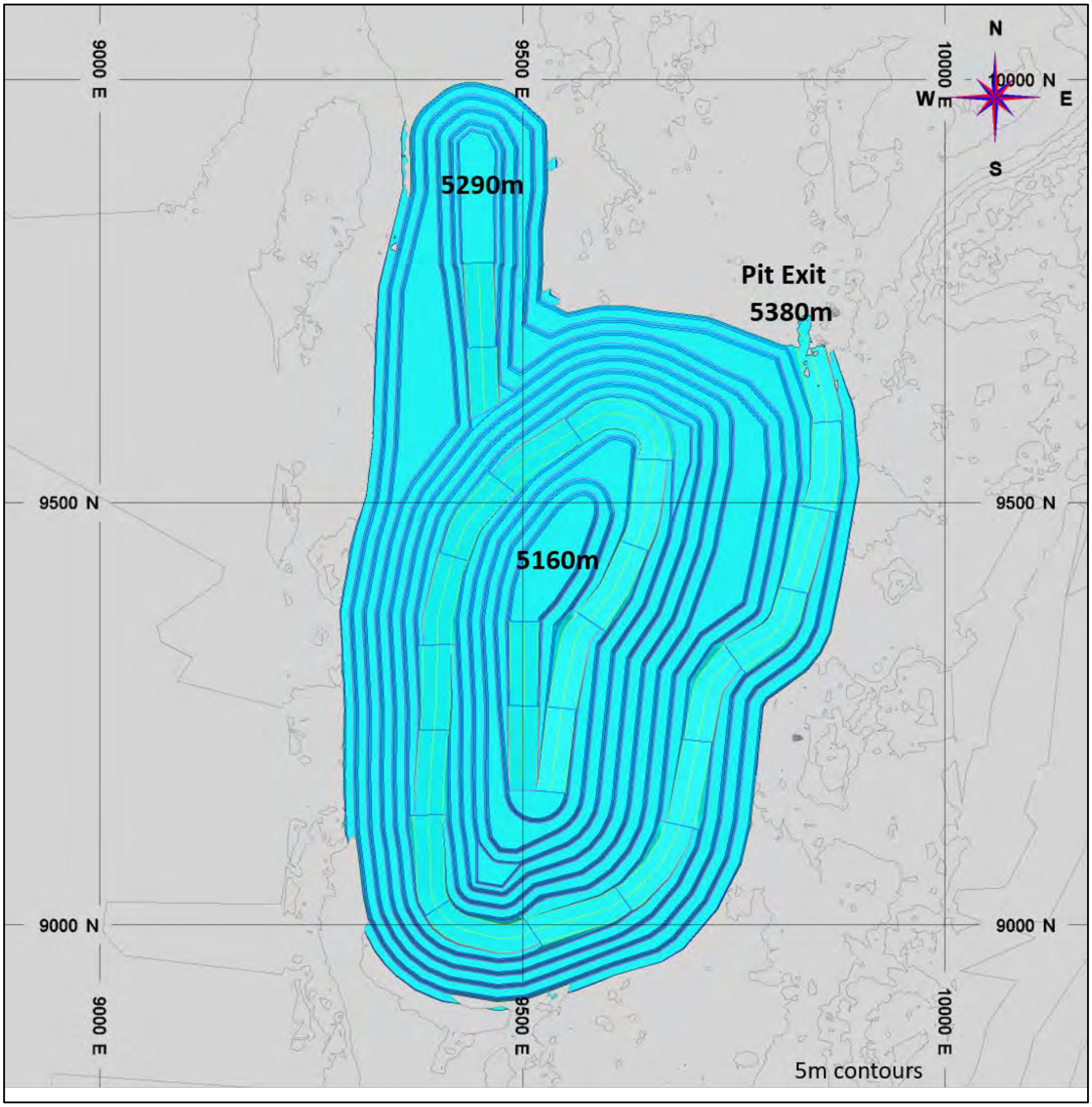
Figure 16-29: SW Phase 1 Design



SW Phase 2

SW phase 2 is the final phase mined in the SW pit. Phase bench elevations range from 5390 masl down to 5160 masl. All waste and mill feed accesses will be on the north side of the pit similar to phase 1. The design is shown in Figure 16-30.

Figure 16-30: SW Phase 2 Design



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. A separate overburden storage facility has been designed for each of the three 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 and Z87 pits. This material was treated as overburden in the PEA so that slopes and mining costs would be appropriate, however, future studies should distinguish the historic waste from overburden for storage destinations. The total amount of waste within the mine plan is 591 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. The lift height of 10 m was used for all storage facilities. Assuming a 36° face slope, the overall slope will be 26.5° with 6.2 m berm widths. A 36° face slope was also used for the backfill slopes into Z87 pit. The final backfill configuration had all of the Z87 exposed pit slopes being covered with waste material, but a volume will remain open in the center of the backfill area.

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.

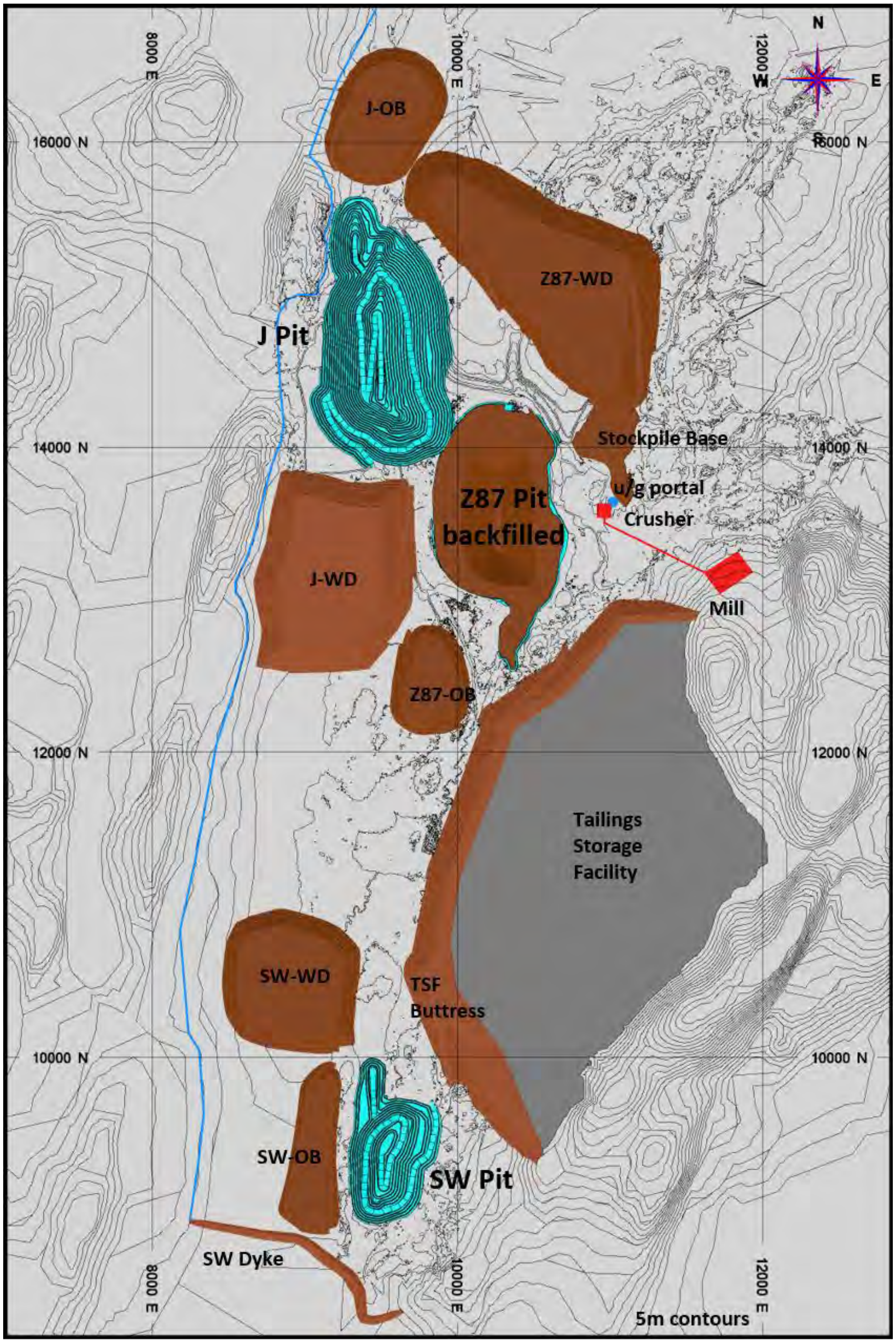
The capacities and top lift elevations for the waste storage facilities are displayed in Table 16-13.

Table 16-13: Waste Storage Facilities Summary

Waste Storage Facility	Storage Capacity (Mm^3)	Top Lift Elevation (masl)
Stockpile Base	1.1	5390
TSF Buttress	39.0	5440
J Overburden	14.5	5400
Z87 Overburden	6.4	5400
Z87-WD	49.7	5440
J-WD	41.0	5420
Z87 Backfill	105.1	5365
SW Dyke	0.8	5395
SW Overburden	4.2	5400
SW-WD	30.3	5430
Total	**292.1**	

The waste destinations are displayed in Figure 16-31. The intent of the SW waste dump is to divert water from entering the SW pit and then allow it to flow in a diversion ditch along the west side of the property.

Figure 16-31: Waste Destinations



16.3.6 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 200 mm bit. This provides the capability to drill 10 metre bench heights in a single pass.

The primary loading units will be 22 m³ hydraulic shovels. Additional loading will be completed by 23 m³ 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 181 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. 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.

16.3.7 Blasting and Explosives

Blast patterns are the same for feed and waste material. The blast patterns will be 6.2m x 5.4m (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 patter size will be 0.30 kg/t. Only emulsion explosives will be used due to the expected wet conditions.

16.3.8 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 10m x 5m 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 20m x 10m with sampling over 6 metres. 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.

16.4 Underground Mining

16.4.1 Introduction

Prior to commencement of the PEA study the following trade off studies were undertaken by AGP:

- Mining Methods Trade Off Study

- Materials Handling Trade Off Study

Mining Methods Trade Off Study

The Troilus deposit is a large, low grade resource. High grade, selective, and high cost mining methods are inappropriate for this deposit. High tonnage, low cost mining methods offer the only economically attractive solutions. A number of differing underground mass mining methods were assessed for suitability for application to the low grade deposit at Troilus. AGP assumed that large-scale mobile equipment will be used in any method selected thereby maximizing productivity and minimizing operating costs.

The mining methods considered comprised:

- Open Stoping with Permanent Stable Rib and Crown Pillars with no backfill
- Slot and Mass Blast with introduced waste material
- Primary and Secondary Transverse Stoping with Cemented Rockfill
- Primary and Secondary Transverse Stoping with Narrow Pillar and Uncemented Rockfill
- Primary and Secondary Transverse Stoping with Wide Pillar and Uncemented Rockfill
- Sub-Level Caving with introduced waste material

A high-level economic model was created for each of the mining methods considered.

In each analysis, a consistent resource area below the planned open pit was selected. Stope dimensions were input in line with geotechnical advice for a single mining method “unit” (e.g. slot stope, rib and crown pillar for Slot and Mass Blast or a primary and secondary stope with narrow pillar). The physicals for each method unit were then estimated by calculation.

Each mining method design was then applied throughout the consistent resource area considered and the total life of mine potential mill feed tonnage and grade was estimated by calculation.

An annual production rate equal to 40 vertical metres of the recoverable resource (mill feed) was calculated for each method.

An existing first principles cost model was adapted to estimate a series of direct unit costs (applied to method physicals) and indirect overhead costs (applied as a cost per day) to estimate a mine operating cost. Waste footwall development adjacent to the mining method unit was included in modelling but capital development and construction costs for such items as ramps, ventilation, rockfill and ore pass raises, materials handling facilities and equipment fleet replacement costs were all excluded.

A net cash flow was then developed for each mining method over the Life of Mine. The dilution and recovery factors were applied to the targeted resource base to generate a net revenue. The operating and sustaining costs were then subtracted to yield a net cash flow. A net present value of the net revenue stream was used to rank the economic potential of each method.

The economic indicators described above only formed part of the selection process for the most appropriate mining method. AGP also sought to describe the positive and negative physical issues associated with each of the mining methods.

Following completion of the trade off study AGP recommended that Slot and Mass Blast ("S&MB") be adopted as the primary mining method. S&MB had the highest economic ranking followed by Sub-Level Caving method ("SLC")

Materials Handling Trade Off Study

The depth and high production rate for the Z87 deposit make trucking of all mill feed and waste to surface economically unattractive. AGP therefore focussed on the following alternative materials handling options:

- Shaft Hoisting System
- Conventional Inclined Conveyor System
- Vertical Conveyor System
- Rail-Veyor System

AGPs methodology was to complete a mine access design suited to each materials handling option to the base of the Measured and Indicated resources. Common development rate assumptions were then used to schedule each option on a quarterly basis, including identification of handling system commissioning time and costs. The same first principles mine cost modelling system described in the mining method trade-off study was used to estimate the capital development costs specific to each of the materials handling options. It was assumed that capital development prior to system commissioning would be undertaken by contractors and that all ongoing development and production would subsequently be undertaken by the owner.

A common production schedule based on the slot and mass blast mining method was applied to each system economic analysis, the start of which was adjusted to suit the materials handling system commissioning dates for each system. The modelled operating cost unit rates used to assess the mining methods were then applied to the production schedule, including estimated operating development requirements. General mine overhead costs were also applied. A specific cost was developed to estimate the operating cost of each handling system as an overhead.

The mine capital development and operating costs of each mine layout schedule were combined with system purchase and installation capital costs, non-mine operating costs and estimated revenues to form a common-base cash flow analysis. Manufacturer budget quotations were received for the vertical conveyor and Rail-Veyor options. The shaft and inclined conveyor capital estimates were derived from recent estimates from other Canadian projects. The resulting net present values from this analysis were then used to create an economic ranking of each materials handling system. In this way, the varying production start dates of the various materials handling systems influenced the economic ranking of the systems.

The economic analysis excluded such items as mobile equipment, other mine infrastructure, process plant and tailings and surface capital expenditure which are deemed common amongst all options to support the mine plan. AGP however believes that the analyses provide a reasonable economic ranking result for each system considered.

The materials handling study indicated that the Rail-Veyor option provides the highest ranking and lowest up-front financing requirements of the four material handling system options considered. The flexibility of design of the system is a particular advantage in following the geometry of the resources. Once established, extension of the system to access deeper resources is simpler, and most likely cheaper, with Rail-Veyor than any of the other options considered.

The Rail-Veyor materials handling arrangement as designed during the materials handling trade off study is shown schematically in Figure 16-32, and the major Rail-Veyor components are illustrated in Figure 16-33. Subsequent to the completion of the trade off study, the PEA work incorporated Inferred Mineral Resources for consideration. This resulted in a 150m extension of the overall strike extent of the deposit under consideration and a 180m increase in depth, resulting in the incorporation of two additional levels of S&MB stopes and associated infrastructure at depth.

The Rail-Veyor system is an innovative rail-based remotely-controlled haulage system which consists of a number of individual trains running on light rail track. Each of the Rail-Veyor trains at Troilus will comprise a number of 2.4m long cars, with each car connected to the next, forming a continuous but articulated 151m long, 30” wide open trough. A total of seven trains will eventually be required to move the required tonnage from the deeper extents of the mine to surface, with each train purchased and placed into service over time as the schedule required. Steel plates mounted to the outside of the cars provide a contact surface for the drive stations to propel the cars along the track. Propulsion is provided by fixed drive stations located at approximately 35m intervals along the inclined track route and consist of electric motors mounted to foam-filled tires that press against the rail car side rails and propel the trains either forward or reverse. Variable frequency drives are used to control the speed of each train, and drive stations are only operating when an approaching train is detected. Dumping of cars on surface is accomplished by way of a vertical dump loop where the cars are inverted, discharging their contents into a bin.

The configuration of the system is a single-track shuttle system with trains running forward and reverse along the track, with numerous bypass sidings incorporated into the system to allow upward-traveling full trains to pass downward-traveling empty trains as illustrated in Figure 16-34. A total of seven 151 m long Rail-Veyor trains will be in operation at Troilus when the mine is producing from the lowest production level. Although this technology is relatively new, two comparable underground Rail-Veyor systems are currently in operation in North America.

Figure 16-32: Rail-Veyor Materials Handling Schematic

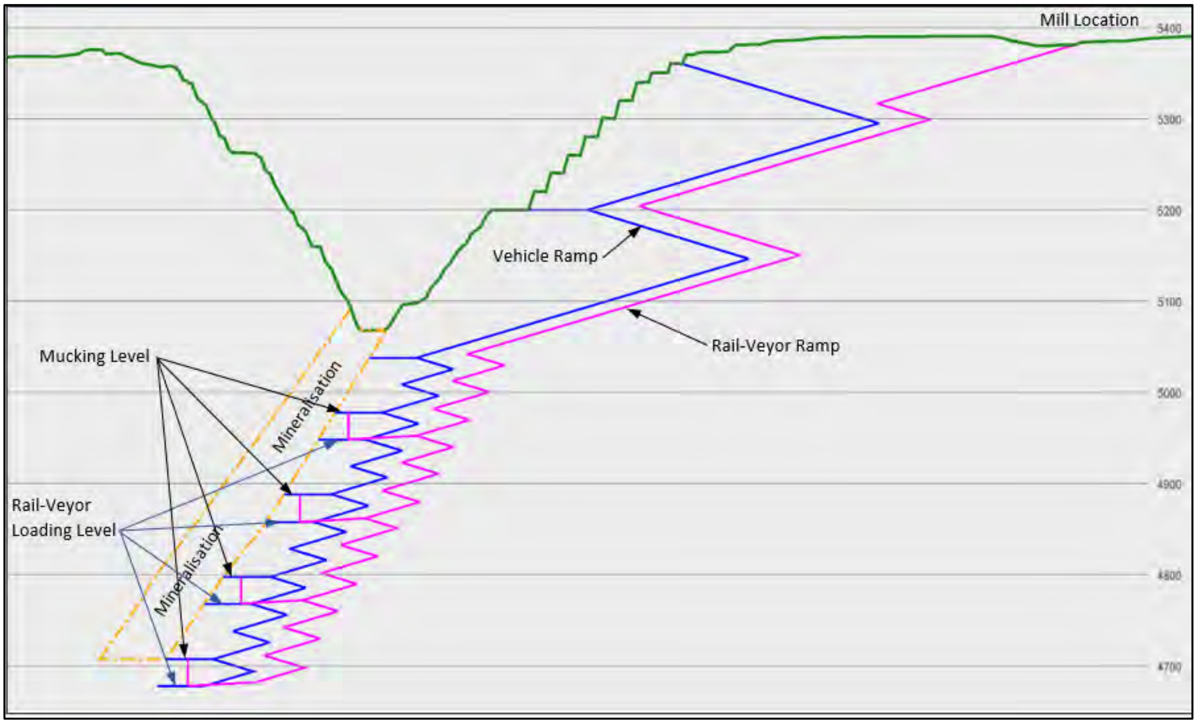


Figure 16-33: Main Rail-Veyor Components

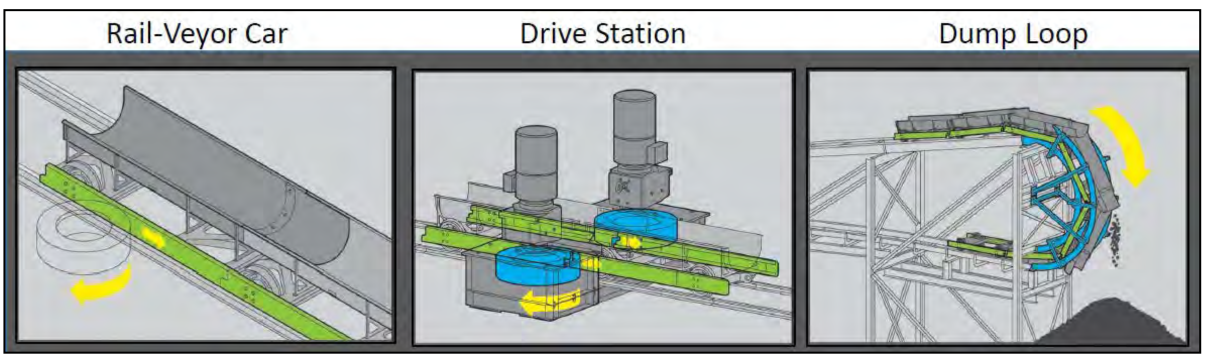
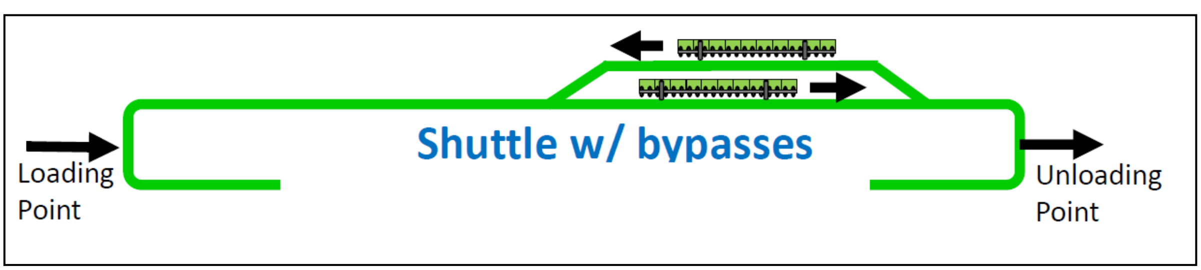


Figure 16-34: Rail-Veyor Shuttle System Operation



The open trough nature of the Rail-Veyor trains allows for the handling of relatively large muck (600 mm or 24 inch top size) through the system, such that the need for an underground crusher to size the material prior to loading the Rail-Veyor system is not required. Instead, the crusher will be located on surface at the mill site to cater for the crushing of both open pit and underground mill feed material, eliminating the need for separate open pit and underground crushers. The two operating Rail-Veyor systems previously mentioned do not use a crusher to size material prior to the Rail-Veyor loading point, but rather use a much simpler rockbreaker arrangement for muck sizing. A similar arrangement is also planned for Troilus.

With reference to the system schematic shown in Figure 16-13, the elimination of underground crushing from the Rail-Veyor materials handling systems allows for the loading of Rail-Veyor trains from each of the stope mucking levels adopting a relatively simple grizzly/rockbreaker/ore bin arrangement. Each mucking level will have two such arrangements located across the strike extent of the deposit. As such, the Rail-Veyor system only has to be installed down to the deepest currently active stoping level at any particular point in time, with deeper Rail-Veyor installations deferred until required in the production schedule.

The main Rail-Veyor ramp is eventually developed down to the full depth of the mine, with a separate rail 'spur' developed from it to service each of the S&MB mucking levels, paralleling the footwall drift required on every second level for stope drilling access purposes. The relatively tall overall height of the S&MB stopes (90 vertical meters) result in the need for only five Rail-Veyor spurs to be developed over the life of mine. Automated track switches will direct trains to the desired levels. As a result, diesel trucking of production stope material from the lower mine has been largely eliminated due to the ability of the Rail-Veyor system to service all stoping areas.

The planned 6.0m average width of the Rail-Veyor ramp will allow for maintenance vehicle access along the entire length of the Rail-Veyor system. This will also incorporate the extra width required for the numerous bypass sidings required. A separate vehicle access ramp will be provided from surface to access all the mine levels for all vehicle, consumables, and personnel movement requirements. A Rail-Veyor car loading arrangement located at the bottom of each rock bin comprising a continuous loader has been included in designs and costing. The Rail-Veyor dump loop illustrated in Figure 16-34 is located on surface at the crest of the portal box cut at the process plant area, and will allow for the discharging of mill feed and waste; waste will be discharged onto the ground adjacent to the dump loop for removal to the waste dump by truck, whilst mill feed will be dumped into a bin and transferred to the surface crusher and then onto the grinding circuit.

16.4.2 Mining Methods

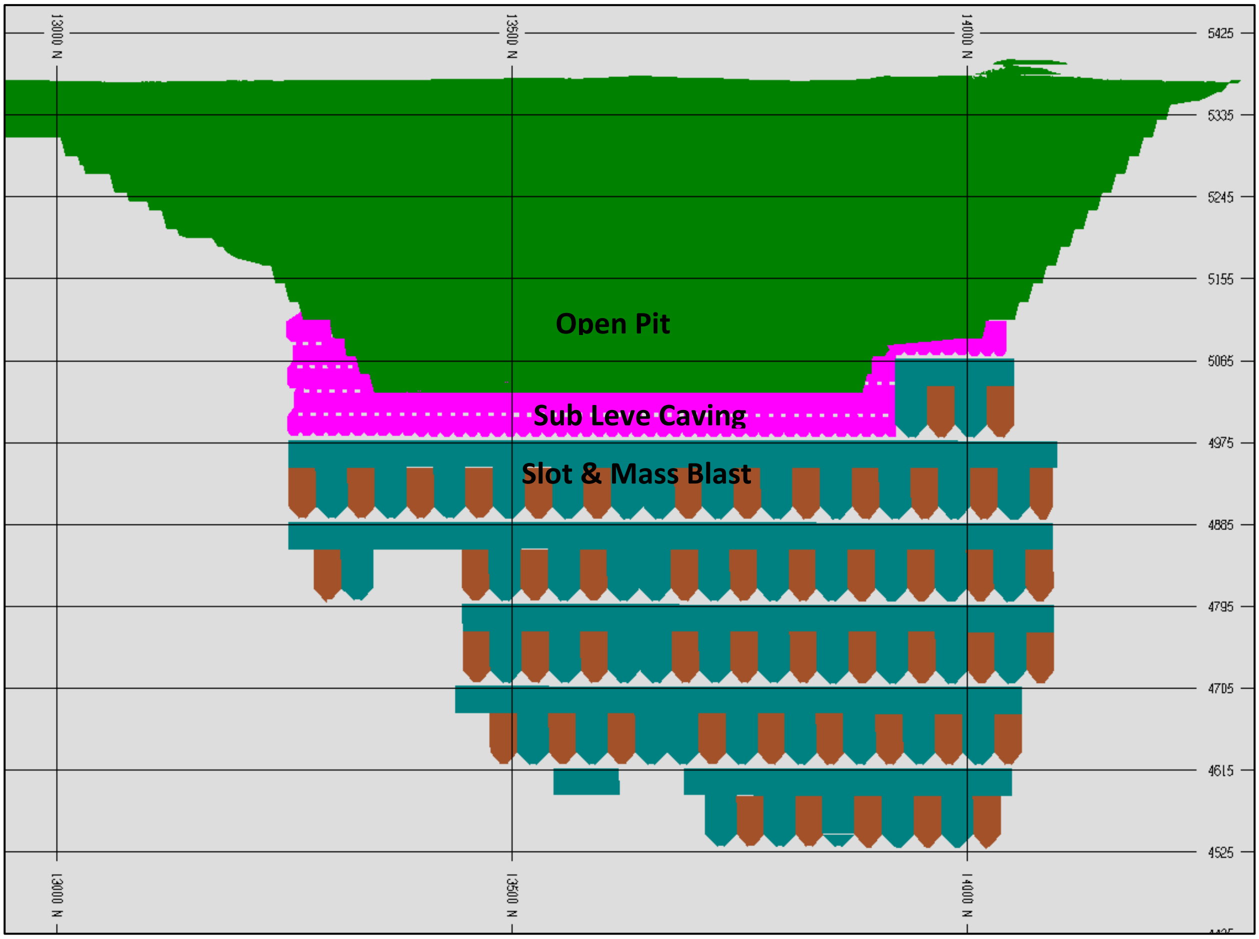
The planned underground mining area is an extension of the Z87 deposit previously mined by open pit at Troilus. Historic mining of the open pit ceased at an elevation approximately 300 m below surface and is now planned to be extended by open pit methods to approximately 350 m below surface. The currently identified Measured, Indicated and Inferred Resources extend to around 900 m below surface and measure a maximum of approximately 850 m along strike. The dip of the orebody varies from around 60^0 to around 40^0, averaging 55^0 in the north and central areas with the flatter dip to the south. Higher grade mineralized areas bifurcate in certain areas, but low grade intervening mineralization allows for the mining of the full section from footwall to hangingwall at satisfactory grades. Stopes vary in thickness up to 80 m true thickness with the thickness generally reducing with depth.

Detailed geotechnical information is provided in Section 16.2 but, in general, ground conditions are considered to be good to very good with strong rock throughout the footwall, orebody and hanging wall sequences. Geological structure in the form of faults and low-angle, widely spaced joints were identified in the exposed open pit sidewalls.

Slot and Mass Blast will be the primary underground mining method used to exploit the Z87 deposit below the open pit floor and will provide 89% of the life of mine underground feed to the mill. The remaining 11% of underground mill feed will be mined using the sub level caving (SLC) method, which is located in the upper portion of the underground mining area, between the deepened open pit and the upper-most level of slot and mass blast stopes. Both of the selected mining methods - as well as the development and operation of the Rail-Veyor materials handling system - operate in a 'top-down' fashion, thus minimising and deferring the mine development necessary to place the mine in operation and sustain production over the life of mine. Initial production will be by SLC during a preparatory phase, followed by S&MB for the bulk of the deposit.

Figure 16-35 shows the areas where each mining method will be applied.

Figure 16-35: Long Section Showing S&MB and SLC Stoping Areas



Slot and Mass Blast

The Slot and Mass Blast (S&MB) mining method is a highly productive bulk mining method, and shares stoping practices commonly found in longhole open stoping, block caving, sub level caving, and other mass draw stoping methods.

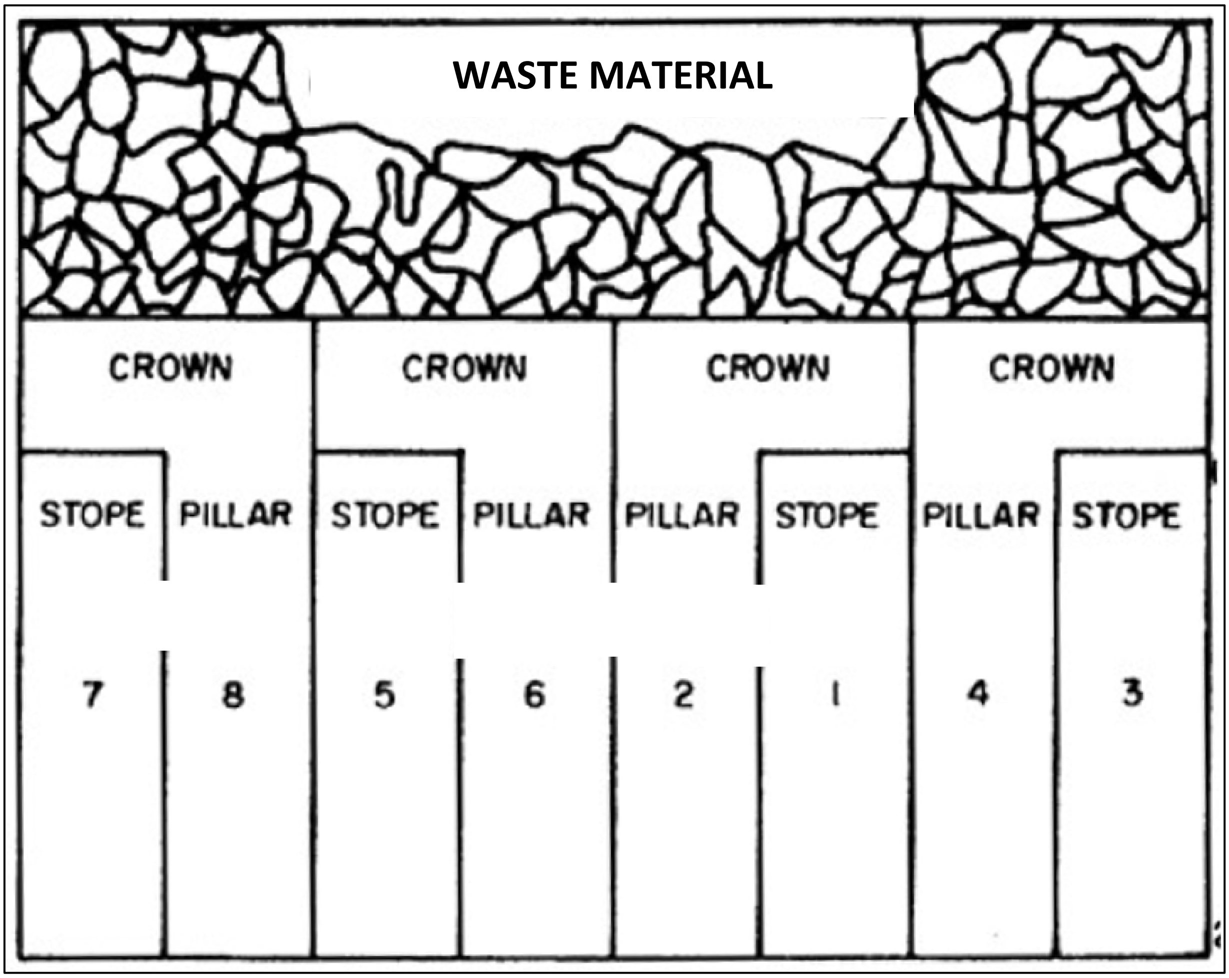
Whilst S&MB is not a common mining method, it has been used successfully at large-scale mining operations in Canada and worldwide. S&MB appears to have been first used at several mines in Canada in the 1960's and 70's, and more recently in Australia and elsewhere worldwide.

In essence, the S&MB mining method is comprised to two distinct elements in each S&MB 'stoping unit' as illustrated in Figure 16-35; the large longhole stope (the 'slot' stope), and the protective pillars to separate the longhole stope from the adjacent stope and the caved waste material above. Each of the Troilus S&MB stopes target the full width of the mineralised zone from hangingwall to footwall, thus only a single rib pillar and overlying crown pillar are required to protect the slot stope from the inflow of waste material from above and from the side as stoping progresses.

The initial slot stope is blasted as large as geotechnically practical and mucked clean with high mining recovery and minimal dilution. The protective rib and crown pillars surrounding the now-open stope are subsequently blasted as a single mass blast such that the pillar muck completely swells into the void created by the combined open stope and mass blast. Stope mucking is accomplished via a trough undercut and herringbone drawpoint arrangement under both the 'slot' and 'mass blast' portions of the stope. As the blasted material from the mass blast is drawn down through the drawpoint system, waste material follows behind - in a similar fashion to the flow mechanics of caved waste in the block caving mining method.

Figure 16-36 shows an illustrative long section of the S&MB method.

Figure 16-36: Illustrative Long Section of the Slot and Mass Blast Method



The extraction sequence for S&MB stoping is as follows:

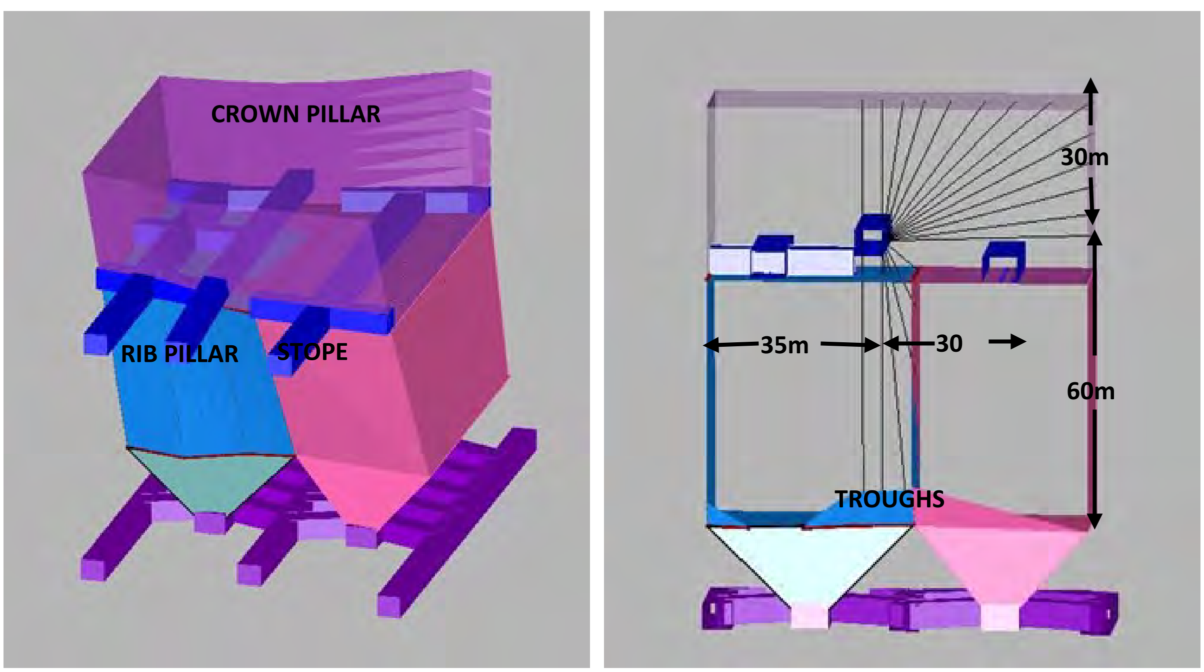
1. Drawpoint trough undercut at bottom of slot stope drilled and blasted
2. Body of slot stope drilled and blasted and mucked empty

3. Drawpoint trough undercut at bottom of rib pillar drilled and blasted
4. Rib and Crown pillars drilled and blasted as a single mass blast
5. As mucking of the mass-blasted mill feed material progresses, waste material from above progressively replaces the mass blast material until the entire void created by the S&MB stope is filled with waste material
6. S&MB stoping can then commence on the neighbouring S&MB stope on the level, with the mass blast stope able to be taken after the previous mass blast firing of the neighbouring stope
7. S&MB stoping could also commence on the S&MB level directly below the S&MB stope on the level above, once mucking of the mass blast was completed

The creation of slot stopes as large as practical and the subsequent mass blasting of the protective pillars into the full void created result in the design of large S&MB stopes. Slot stope dimensions are 30m along strike, 60m high (inclusive of trough undercut), and full orebody width. Rib pillar dimensions are similar to slot stopes, but average 35m along strike, whilst the overlying crown pillar is 30m high, averages 65 m along strike and full orebody width. Given these stope dimensions, mass blast tonnages at Troilus range from 250 kt to 870 kt per S&MB stope, with an average mass blast tonnage of 450 kt. Total stope tonnages (slot stope + undercuts + mass blast) range from 370 kt to 1,300 kt per S&MB stope and average approximately 740 kt per stope.

A typical Troilus S&MB stope design is illustrated in Figure 16-37. The overall S&MB stoping area and arrangement was shown previously in Figure 16-35.

Figure 16-37: Slot and Mass Blast Stope Design



Trough undercut drilling will be completed using 102mm (4 inch) upholes drilled from the mucking level using a top hammer long hole drill. The stope, rib pillar and crown pillar blast holes are drilled 152mm (6 inch) in diameter with an In- the-Hole (ITH) hammer drill employing a combination of upholes and downholes drilled from the base of the crown pillar level shown in Figure 16-18. Typically, holes of up to 100m in length are drilled with this type of drill at other Canadian mines, which exceeds the maximum hole length of around 50m required at Troilus. As such, drillhole accuracy is expected to be good. The ITH drill with a Machines Roger V-30 head will also drill a 700mm easer slot raise between the drill and drawpoint levels to assist initial blasting of the stope.

The very large mass pillar blasts will be technically challenging but have been successful elsewhere. Notwithstanding, fragmentation in the stope will likely be somewhat variable and large blocks will likely occur. Secondary breakage at stope draw points will be required.

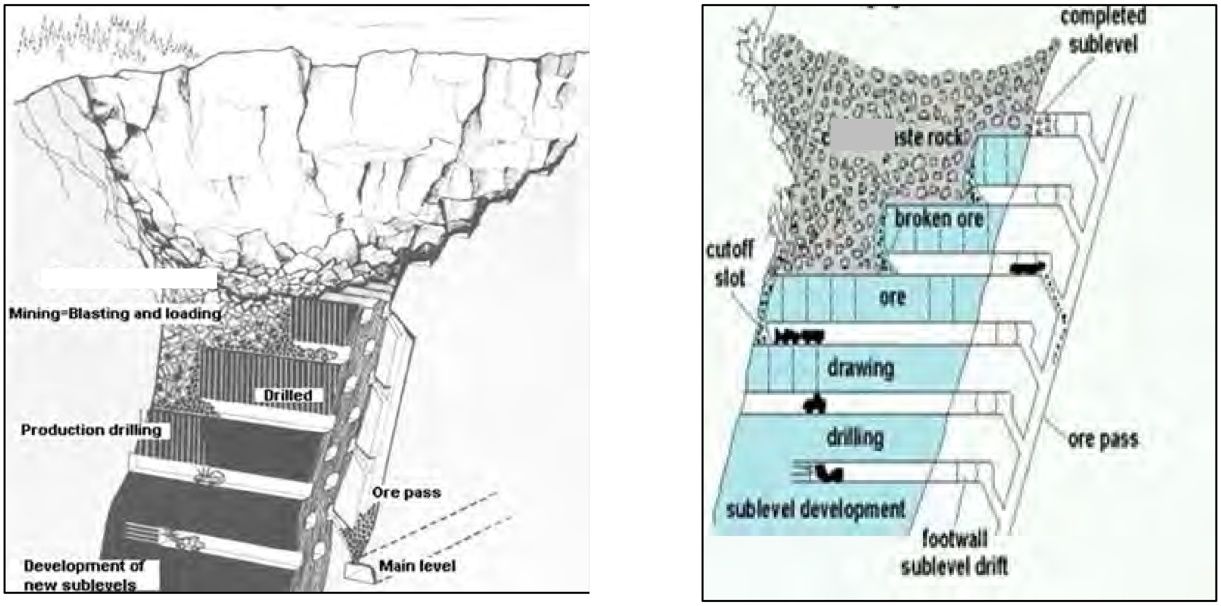
Sub Level Caving

The sub level caving stoping method is a common mining method used at many mining operations worldwide and is illustrated in Figure 16-38.

SLC stoping at Troilus is located between the open pit and the top of the S&MB stopes as shown previously in Figure 16-37, and serves two main purposes:

1. Allow the full strike extent of the upper-most level of the S&MB stopes to be overlain by broken material in order for the mass flow of waste material required by the S&MB mining method to function as required.
2. Create a grade buffer, or 'blanket' of blasted SLC material across the entire top of the S&MB stoping area in order to minimise the migration of waste material reporting to the S&MB draw points and being mucked as millfeed. The grade blanket is created by limiting the mining recovery of the SLC stopes.

Figure 16-38: Typical SLC Stoping Operations



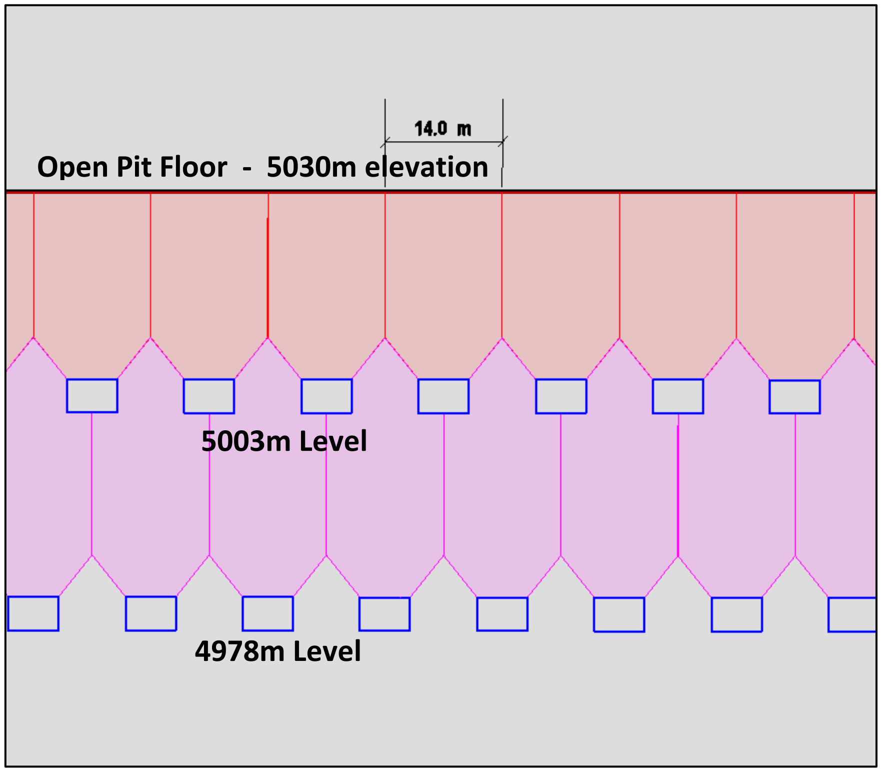
An important aspect of the SLC method is that ring blasting is constrained by broken waste rock following the retreating blast faces. However, because the majority of the upper-most level of the SLC main stoping area is located immediately below the open pit floor, broken waste from extension of the Z87 open pit prior to the commencement of underground mining will be placed along the base and lower pit benches in the open pit to provide the required confinement before SLC mining operations commence. The introduced waste will then follow mining downwards behind the mill feed material as stope mucking progresses through the SLC operation and into the S&MB stoping area. It has been assumed that the amount of introduced waste required to be placed into the Z87 open pit prior to the start of underground production is equivalent to the total tonnage of the SLC operation and the first S&MB stoping level, which is approximately 10 Mt of broken waste material. Significantly more waste material is available from the extension of Z87 open pit if needed.

Ultimately, as underground mining progresses downward through the orebody, the hangingwall country rock will be progressively exposed. Although the hangingwall rock unit is relatively competent, it will ultimately cave and contribute additional broken waste material onto the introduced waste in the open pit.

The Troilus SLC operation itself will be similar to many existing transverse SLC operations. Mill feed crosscut are 6.0 m wide on a 14.0 m centerline spacing, The SLC stopes are mined transversely, with stope crosscuts driven through from the footwall drift to the hangingwall where slot raises are used to initial SLC stoping are located.

Sub level vertical interval will vary from 25m to 27 m. Ring drilling will be completed with 102 mm (4 inch) upholes drilled using the same type of tophammer long hole drill used to drill the S&MB trough undercuts. There is potential for dilution after every SLC ring blast, so higher levels of dilution are to be expected than for S&MB. A typical long section through the two SLC stoping levels located below the open pit floor is shown in Figure 16-39.

Figure 16-39: Typical Long Section of SLC Stoping Immediately Below Open Pit Floor



16.4.3 Cut Off Grade

The study cut-off grade analysis was undertaken after completion of the mining method trade off study at the commencement of the PEA study using the current project parameters identified by AGP and Troilus. These parameters are shown in Table 16-14.

Table 16-14: Project Parameters for Cut-off-Grade Analysis

Description	Unit	Value
Exchange Rate	Can$/US$	1.30
Metal Prices		
Gold	US$/oz	1,350
Copper	US$/lb.	3.00
Silver	US$/oz	18.00
Process		
Copper Recovery	%	90
Cu Concentrate Grade	% Cu	19
Concentrate Moisture	% H2O	8
Gold Recovery	% Au	90
Silver Recovery	% Ag	50
Concentrate Transport		
Truck	Can$/wt.	52
Ship	Can$/wt.	52
Smelter Terms		
Treatment Charge	US$/t	85
Payable Copper	% Cu	96.5
Minimum Deduction Cu	% Cu	1
Payable Gold	% Au	97
Minimum Deduction Au	g/t Au	1
Payable Silver	% Ag	97
Minimum Deduction Ag	g/t Ag	30
Copper Refining	$US/lb.	0.085
Gold Refining	$US/oz Au	5.0
Silver Refining	US$/oz Ag	1.0
Non- Mining Costs		
Process	Can$/t	11.57
G & A	Can%/t	3.25
Royalty	%	0

In Table 16-14 the process metal recoveries are based on previous metallurgical work on a flotation circuit only, with a copper concentrate being the sole product sold to the market. The processing and G&A unit costs are based on a simultaneous co-production scenario with a large open pit tonnage.

The *in situ* resource was investigated using the block model by creating a series of potential mining shapes at a range of cut-off grades from 0.4 to 1.5 eq g/t Au. The mining method trade off study had indicated a base case mine operating cost of Can$ 24.04/t +15% to account for sustaining capital at a production rate of 8,500 tpd was assumed at an *in situ* cut-off grade of 0.8 eq g/t Au. Higher production rates (tpd) were assumed to be constrained by rock handling systems while lower production rate were assumed to be variable with resource tonnage and the mine operating cost set at 65% variable with production rate.

A net revenue calculation was then conducted for each in situ cut-off grade.

The calculation is summarised in Table 16-15.

Table 16-15: Preliminary Production Rate and Mining Cost Assumptions at Range of In situ Cut-off Grades

Description	Unit Value	Unit	Unit	Cut-Off 0.4 eq g/t	Cut-Off 0.6 eq g/t	Cut-Off 0.8 eq g/t	Cut-Off 1.0 eq g/t	Cut-Off 1.2 eq g/t	Cut-Off 1.5 eq g/t
In Situ Resource Tonnes		Tonnes (Mt)		43.7	31.2	22.7	19.8	17.7	11.0
Gold Equivalent Grade		Au Eq g/t		1.23	1.47	1.70	1.78	1.86	2.16
Mining Dilution		%		8%	10%	12%	14%	16%	18%
Mining Tonnage Recovery		%		96%	96%	96%	96%	96%	96%
Millfeed		Tonnes (Mt)		45.3	32.9	24.4	21.6	19.8	12.5
Gold Equivalent Grade		Au Eq g/t		1.15	1.36	1.55	1.59	1.64	1.88
NET REVENUE			**Can$M**	**2,476**	**2,120**	**1,788**	**1,637**	**1,540**	**1,111**
Operating Costs	Base Case								
Production Rate tpd	8,500	tpd		10,000	9,000	8,500	7,500	6,500	4,000
U/G Mining Operating Cost (65% Fixed)	24.04	Can$/t processed		22.78	23.57	24.04	25.16	26.63	33.50
Sustaining Capital	3.61	CDN$/t processed		3.42	3.54	3.61	3.77	3.99	5.03
Mining Operating Cost			Can$M	1,031.2	776.2	586.1	544.5	526.1	417.2
Mining Sustaining Cost			Can$M	154.7	116.4	87.9	81.7	78.9	62.6
Concentrator	11.57	CDN$/t processed	Can$M	523.9	381.0	282.1	250.4	228.6	144.1
G & A	3.25	CDN$/t processed	Can$M	147.2	107.0	79.2	70.3	64.2	40.5
Concentrate Trucking	52.00	CDN$/t concentrate	Can$M	13.2	11.3	9.5	8.7	8.2	5.9
Cu Concentrate Shipping	52.00	CDN$/t concentrate	Can$M	13.2	11.3	9.5	8.7	8.2	5.9
TOTAL OPERATING COST			**Can$M**	**1,883.2**	**1,403.3**	**1,054.4**	**964.4**	**914.3**	**676.2**
Revenue Less Operating Cost			**Can$M**	**592.9**	**716.4**	**733.4**	**672.8**	**625.3**	**434.8**

In order to assess sensitivity analyses with mining costs +/- 10% were also reviewed. Figure 16-40 shows the results of this analysis.

Figure 16-40: Net Revenue versus Mining Costs for a Range of In situ Cut-Off Grades

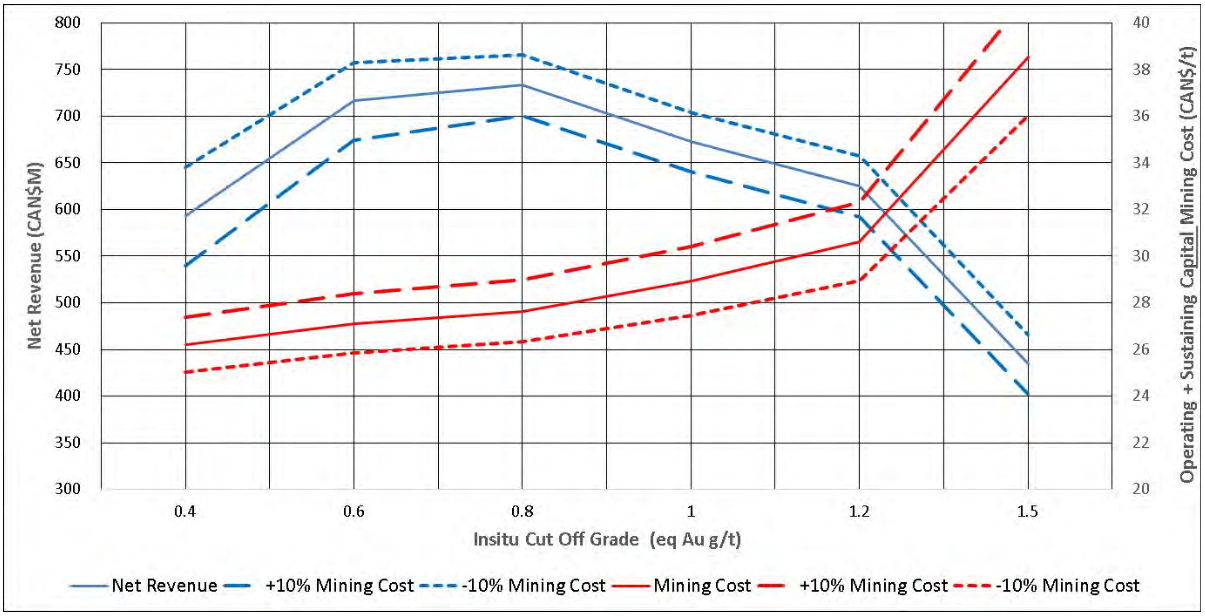


Figure 16-40 shows the maximum net revenue to occur at an in situ cut-off grade of 0.8 eq g/t Au. This target in situ cut-off grade was therefore used to define stope outlines in the block model except that, where bifurcation of higher grade mineralization occurs, and selective mining is not possible, lower grade mineralization was included to increase stope size and maintain low mining costs. Lower grade mineralisation was also added where necessary to improve the extractability of mining shapes.

16.4.4 Application of Modifying Factors

Prior to the commencement of SLC production the base of the open pit will be filled with waste from active open pit mining operations. This waste material will cover the SLC stopes where they day-light into the open pit to confine the SLC stope blasts but will also provide the potential for dilution as the broken ring material is mucked. The mining recovery of broken ring material from the SLC stopes has been reduced in order to minimise dilution and establish a 'grade blanket' between the broken ring material and the introduced waste above. Reducing the mining recovery of the broken ring material will have the effect of improving the mill feed grade from the SLC stopes. A substantial portion of the tonnage contained within the grade blanket can be recovered at the end of life of mine by 'overpulling' draw points from the lowest-most drawpoint level, however it has been assumed for this study that this material will not be recovered and thus not contribute to underground mill feed.

The following modifying factors were applied to the insitu resources in SLC mining areas:

- Development – 100% mining tonnage recovery at 0% dilution
- Ring Extraction – 75% mining tonnage recovery at 15% dilution with zero grade
- For S&MB each stope the insitu resource tonnage and grade was calculated for development, each trough, slot stope and mass blast (rib + crown pillar) based on 3D

design wireframes of each S&MB stope. Modifying factors were applied to each component wireframe as shown in Table 16-16

- Additionally, there is some mill feed grade material between the S&MB trough undercuts that fall outside of the stope design wireframes and is therefore excluded from stope tonnes and grade. This area can clearly be seen in Figure 16-38, and is wholly comprised of above cut-off grade material. In order to ensure that the broken grade blanket and the overlying broken waste is able to flow as required down through the mining voids as mining progresses to deeper S&MB levels, these areas of solid ground will be drilled and blasted prior to the firing of the mass blast of the S&MB stope below. It has been assumed for this study that none of this wrecked tonnage is recovered as mill feed; it simply contributes to the tonnes and grade of the grade blanket, thus helping to improve the mucking grades from the future S&MB stopes on the lower levels of the mine

Table 16-16: Slot and Mass Blast Modifying Factors

Description	Dilution Assumptions	Mining Tonnage Recovery Assumption
Development	0% dilution.	100%
Both Troughs and Stope	3.4% dilution at 20% insitu grade (to account for 1.0m of dilution from the hangingwall and footwall stope contacts in a stope of average thickness.)	94%
Mass Blast	4.1% dilution at 20% insitu grade (to account for an additional 1.0m of dilution from the hangingwall and footwall stope contacts of both the stope and mass blast area in an area of average thickness.). Plus 10% dilution at zero grade (from open pit waste intermingling with resource material during blasting and mucking)	80%

The resulting estimate of total life of mine mill feed from the mining method components is shown in Table 16-17. The classified resource source of life of mine total mill feed is shown in Table 16-18.

Table 16-17: Components of Total Mill Feed

Description	Unit	Value
SLC		
SLC Stope	kt	4,827
SLC Dev	kt	472
Total SLC	kt	5,299
Eq Au	g/t	1.38
Au	g/t	1.18
Cu	%	0.13
Ag	g/t	1.33
S&MB		
Mass Blast Body	kt	22,497
Mass Blast Undercut	kt	2,170
Stope Body	kt	7,852
Stope Undercut	kt	1,790
Total S&MB	kt	37,017
Eq Au	g/t	1.34
Au	g/t	1.18
Cu	%	0.10
Ag	g/t	0.79
Total Millfeed	**kt**	**42,316**
Eq Au	g/t	1.35
Au	g/t	1.18
Cu	%	0.11
Ag	g/t	0.86

Table 16-18: Source of Total Mill Feed by Resource Classification

Resource Classification	Tonnes kt	Eq Au g/t	Au g/t	Cu %	Ag g/t
Indicated Resources	30,106	1.34	1.17	0.11	1.08
Inferred Resources	12,210	1.36	1.21	0.09	0.33
Total	**42,316**	**1.35**	**1.18**	**0,11**	**0.86**

16.4.5 Underground Mine Design

The Z87 underground mine consists of two levels of SLC stoping immediately below the floor of the open pit - with minor amounts of SLC stoping up along the open pit side walls - and five 90m high S&MB stoping areas located below the SLC stoping levels as illustrated in Figure 16-36. All stoping levels as well as all other mine infrastructure will be accessed from the footwall side of the deposit.

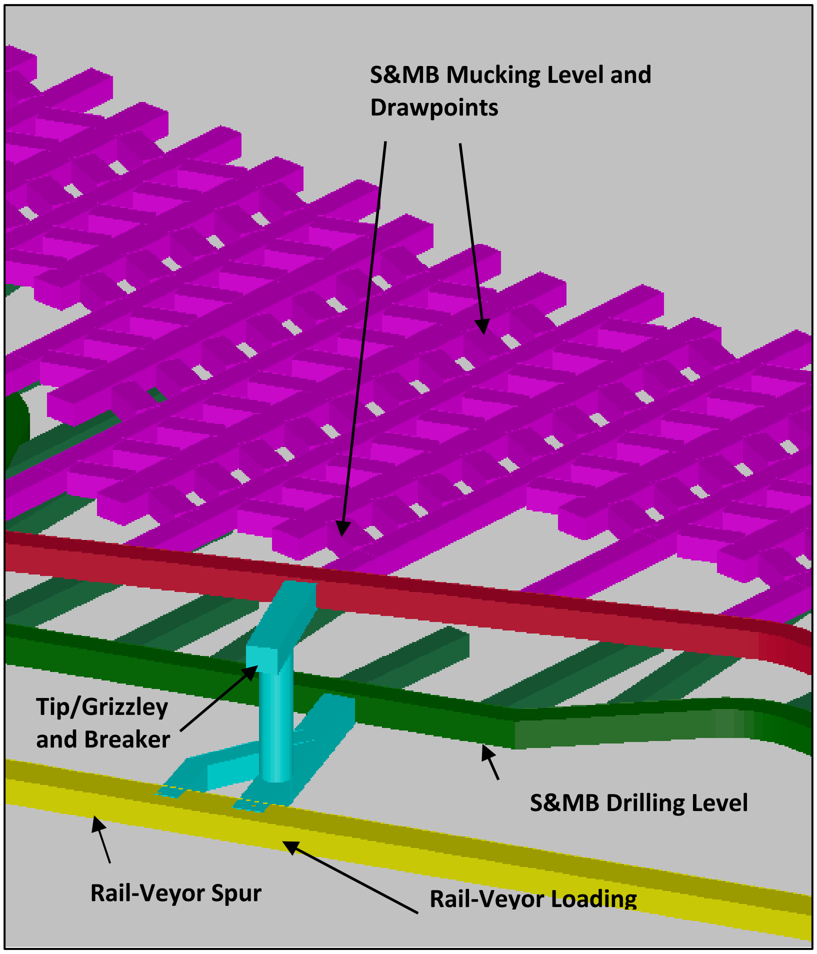
The SLC and S&MB stopes are mined transversely from footwall drifts. With respect to the S&MB stopes, mucking access is provided at the base of the stope via a herringbone arrangement of draw points under both trough undercuts of each S&MB stope. The mucking level is also the drilling location for the trough undercuts. Stope drilling access is located 60 m above the drawpoint level, at the base of the S&MB crown pillar. All slot stope drilling as well as rib and crown pillar drilling is conducted from this level.

The S&MB stope drilling level is also the location of the Rail-Veyor loading level for the stope above, whose drawpoint level is located 30m above. A rock bin is located between the mucking level of the stope above and the drilling/Rail-Veyor level of the stope below, which provides a dump point and surge capacity between stope mucking and Rail-Veyor car loading. Each rock bin is equipped with a grizzly and rockbreaker in order to limit the material top size to the Rail-Veyor cars to -24". Two rock bin arrangements are provided along the strike length of each S&MB mucking level to ensure efficient LHD productivity from stope to rock bin. A typical rock bin arrangement is illustrated in Figure 16-41.

Separate ramp systems have been designed for personnel/vehicles and the Rail-Veyor materials handling system. The main ramp is collared on an upper bench of the open pit, and provides access for all personnel, mobile equipment and supplies to each footwall drift, the access of which is centrally located along strike. In addition, a temporary main ramp access located lower down within the confines of the open pit has been designed to provide early access for underground mine development and waste disposal purposes.

The Rail-Veyor ramp portal is located in a box cut at the mill location and terminates at the bottom-most level of the mine. Both ramp systems run parallel to each other until the elevation of the bottom of the open pit is reached, after which the Rail-Veyor ramp continues the planned route downward at the southern end of the deposit. Rail-Veyor spurs driven from the main Rail-Veyor ramp provide access to each S&MB loading level drift. Each S&MB loading level drift runs the full strike length of the mine in order to service the two rock bin loading arrangements located on each level whilst allowing tail room through the loading points for the 151 m long Rail-Veyor trains.

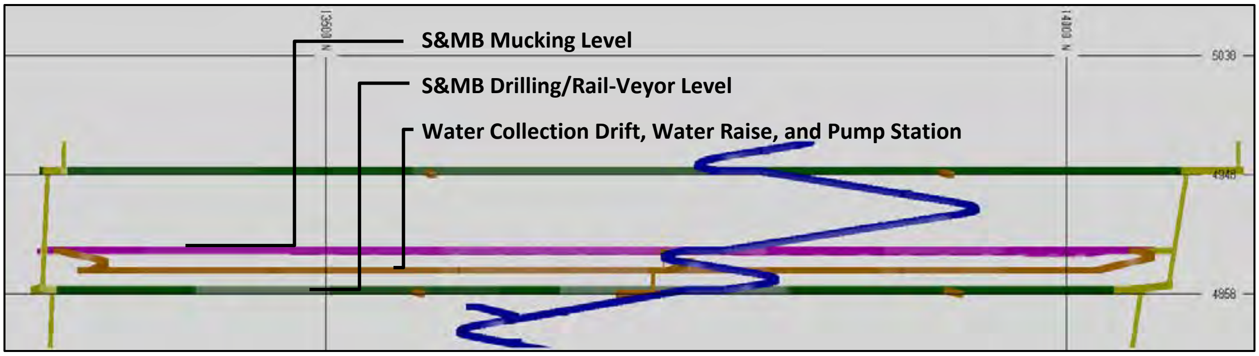
Figure 16-41: Typical S&MB Materials Handling Arrangement



Both the main access ramp and the Rail-Veyor ramp act as ventilation intake airways. Intake air will flow onto each level via the ramp systems and will be exhausted at both ends of each footwall drift via a series of exhaust raises and lateral drifts. The lateral ventilation drifts are required to cater for the dip of the deposit as well as to locate the collar of the surface exhaust raises to the eastern (footwall) side of the open pit rim. An additional centrally-located intake raise system has been provided to facilitate pre-production development at depth prior to the establishment of the primary exhaust ventilation arrangement at depth.

Four main pump stations are located at various elevations through the mine to cater for the large expected water inflows during spring freshet and summer storm events. As such, each pump station will 'service' two levels of S&MB stopes. Each of the dewatering arrangements consist of a dedicated water collection drift located at an elevation halfway between a S&MB mucking level and the S&MB drill level/Rail-Veyor level below and will be developed along the full strike extent of the ore body. Water reporting to the stope draw points above is directed into the water collection drift via a series of large diameter drain holes collared along the strike extend of the drawpoint level. From here, the collected water is directed down a water raise to the pump room and pumped to surface. An example of a typical water collection drift and pump room arrangement is illustrated in long section in Figure 16-42.

Figure 16-42: Typical Water Collection Drift and Pump Room Arrangement



A large underground workshop/office/fuel bay complex is located on the 4948m Level in the upper portion of the mine, adjacent to the northern return airway system. The 4948m Level provides access for S&MB stope drilling and Rail-Veyor car loading and is a suitable low-traffic area for workshop activities. A Rail-Veyor maintenance facility will also be incorporated into the workshop complex. In addition, a satellite fuel bay will be located deeper in the mine later in the life of mine.

Explosives and detonator storage will be located an appropriate distance off of the main ramp access in the upper portion of the mine.

A lunchroom/refuge station will be located at the entrance to each S&MB mucking level.

A typical S&MB mucking level is shown in Figure 16-43 and a typical S&MB drilling/Rail-Veyor loading level in Figure 16-44.

The overall mine design is shown in Figure 16-45 to Figure 16-48, inclusive. A summary of waste (capital) development metres by development type is shown in Table 16-19.

Figure 16-43: Plan View of Typical S&MB Mucking Level

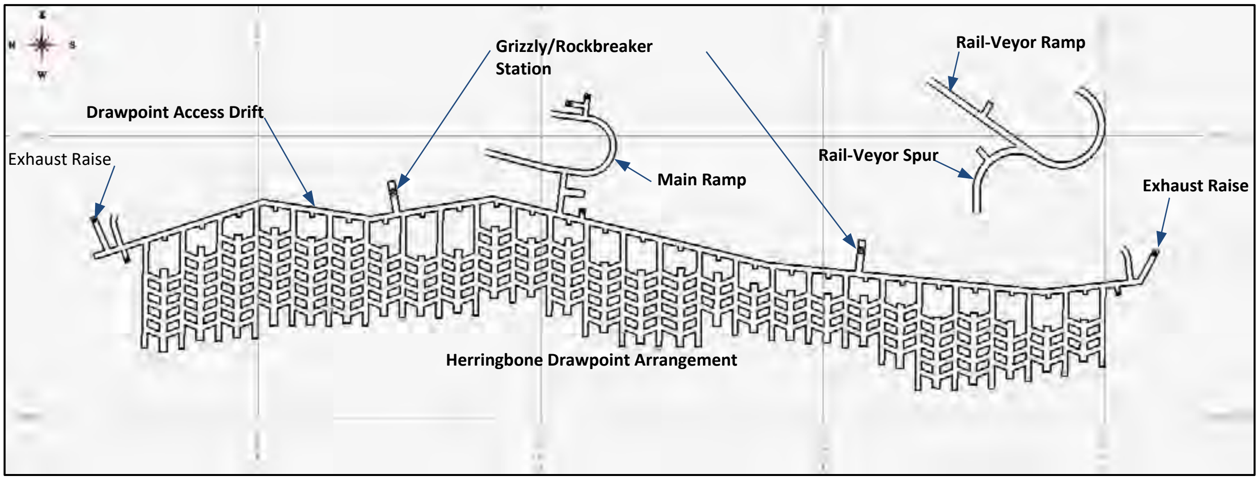


Figure 16-44: Plan View of Typical S&MB Drilling/Rail-Veyor Loading Level

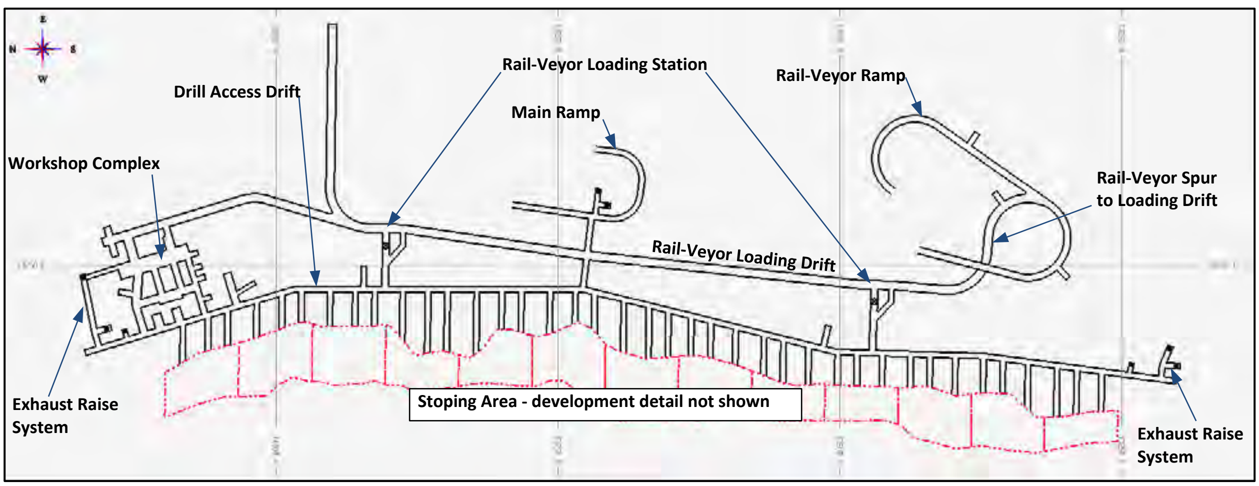


Figure 16-45: Plan View of U/G Mine with Transparent Open Pit

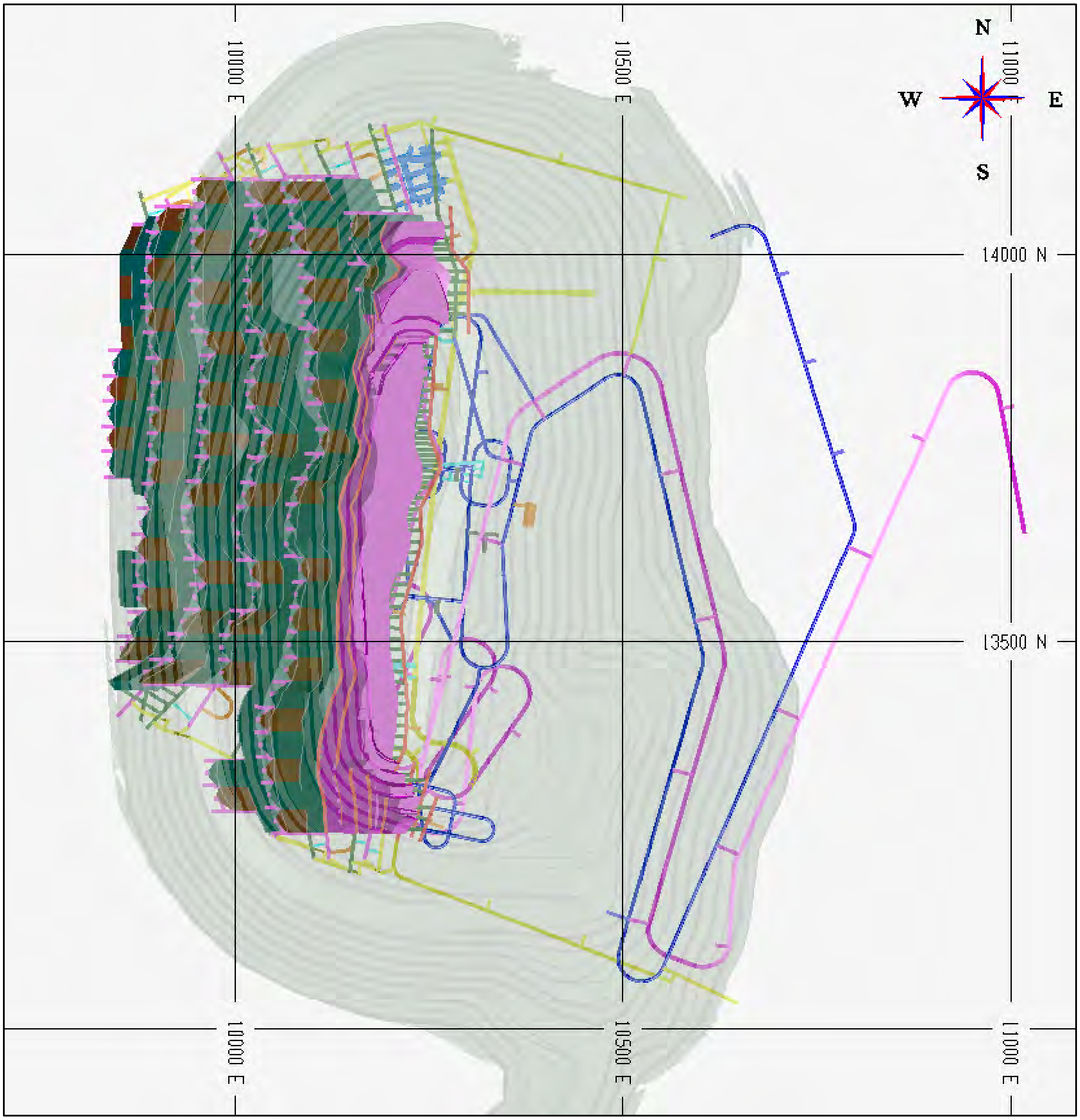


Figure 16-46: Cross Section View of U/G Mine. Looking North

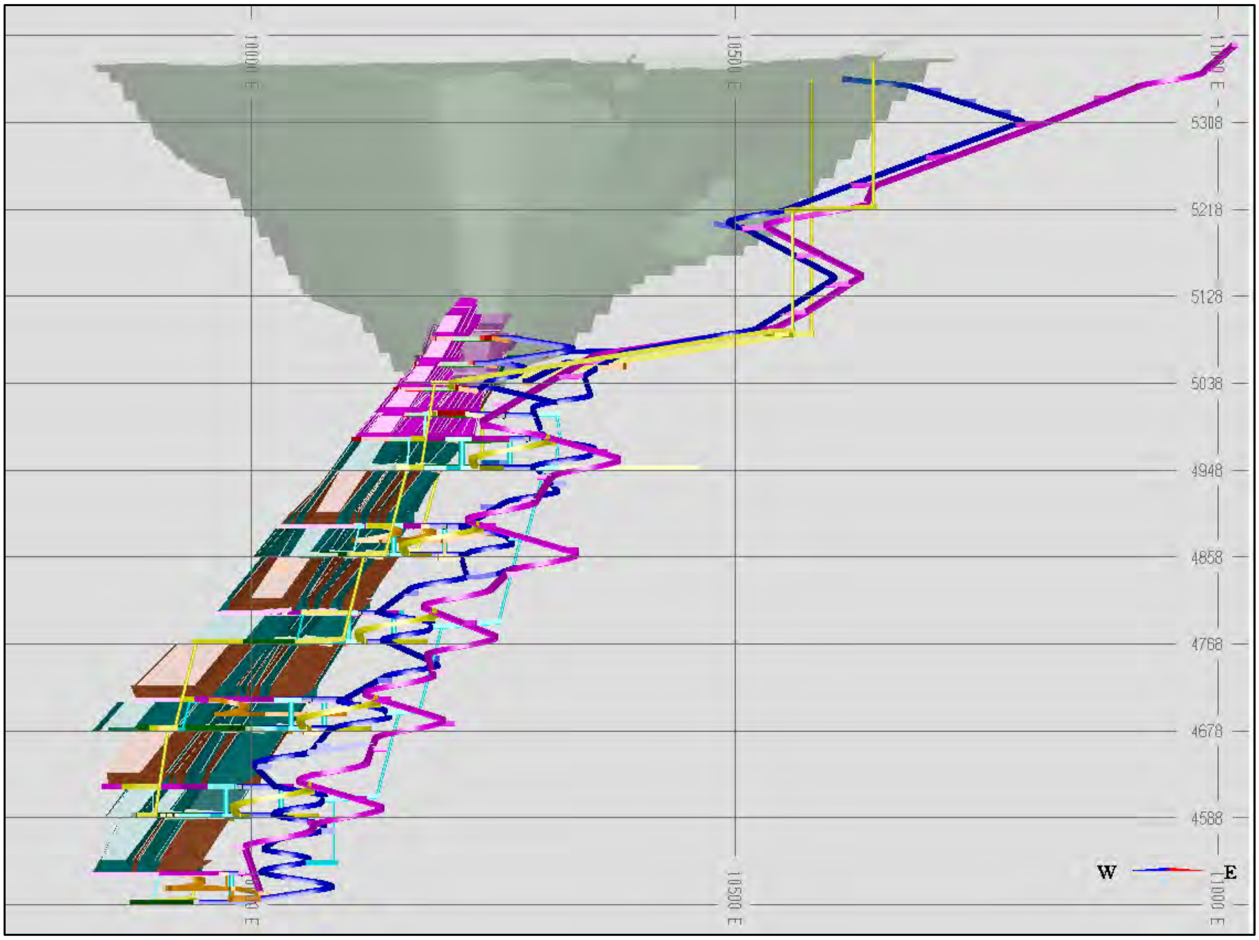


Figure 16-47: Long Section View of U/G Mine. Looking West

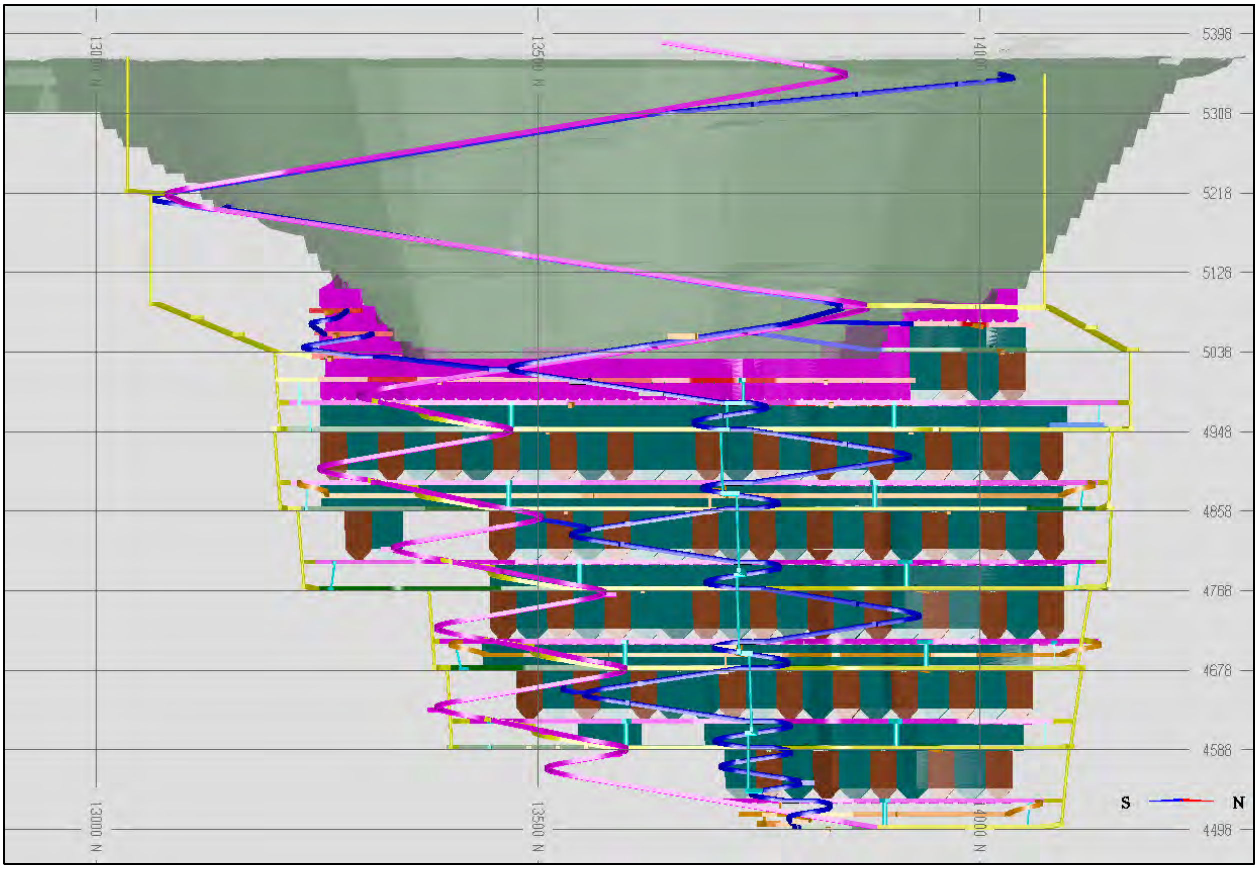


Figure 16-48: Isometric View of U/G Mine. Looking Northeast

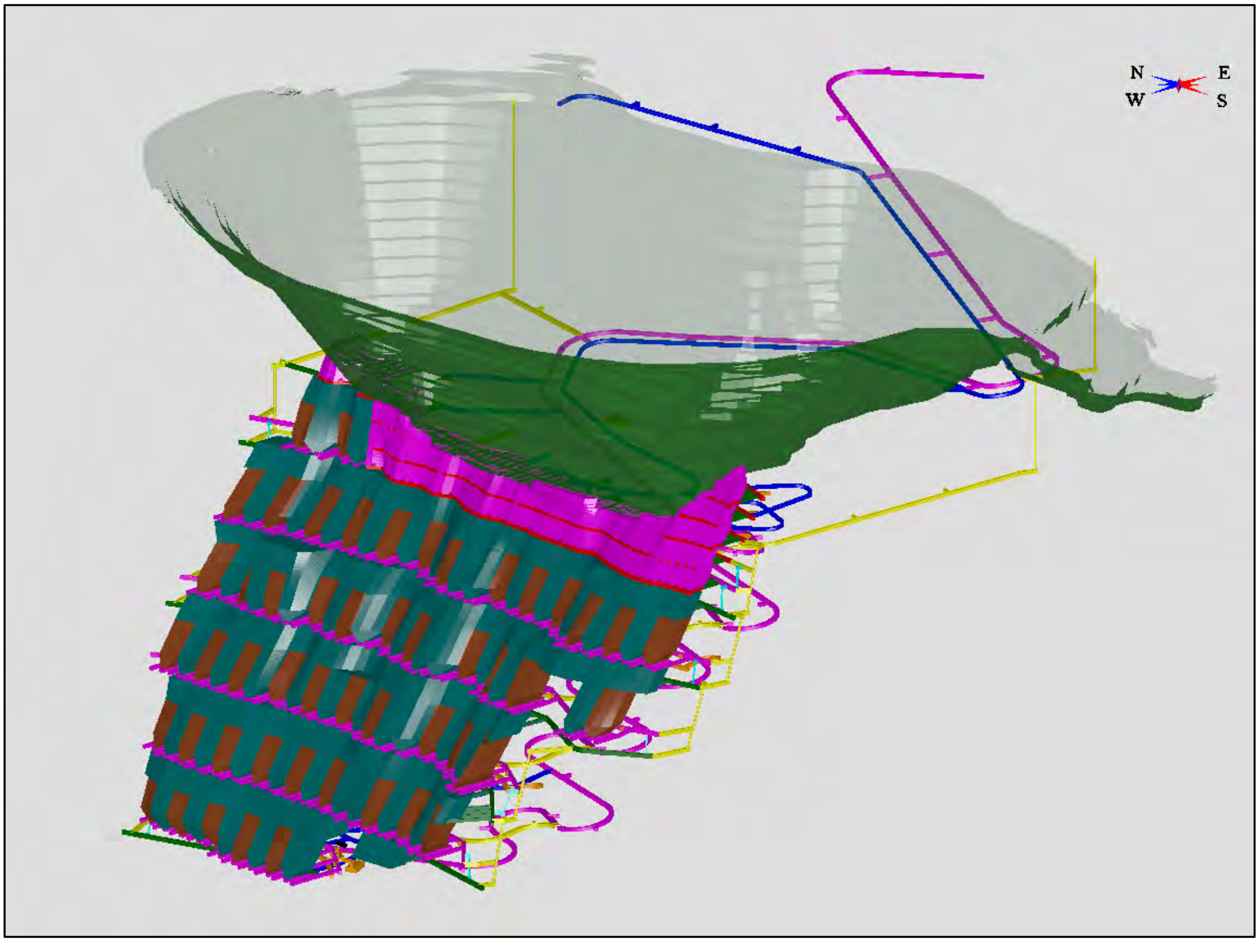


Table 16-19: Waste Development Metres by Type

Waste Development Type	Dev. Metres	
Vehicle Ramp	7,560 m	17%
Materials Handling - Rail-Veyor Ramp	10,460 m	24%
Materials Handling - Rail-Veyor - Other	1,840 m	4%
Level Access	1,340 m	3%
Level	11,130 m	25%
Ventilation	2,930 m	7%
Dewatering	4,050 m	9%
SLC Crosscut access	4,000 m	9%
Orepass/Maintenance/Ancillary	1,150 m	3%
Total	**44,460 m**	

16.4.6 Ventilation

The fresh air demand has been designed to meet the Québec Provincial Regulation Respecting Occupational Health and Safety in Mines (“RROHS”). The ventilation system has been designed to meet the total fresh air requirement at all times throughout the mine. The proposed network ensures that the development and production peaks will be met by zone/level and provides flexibility to the ventilation system.

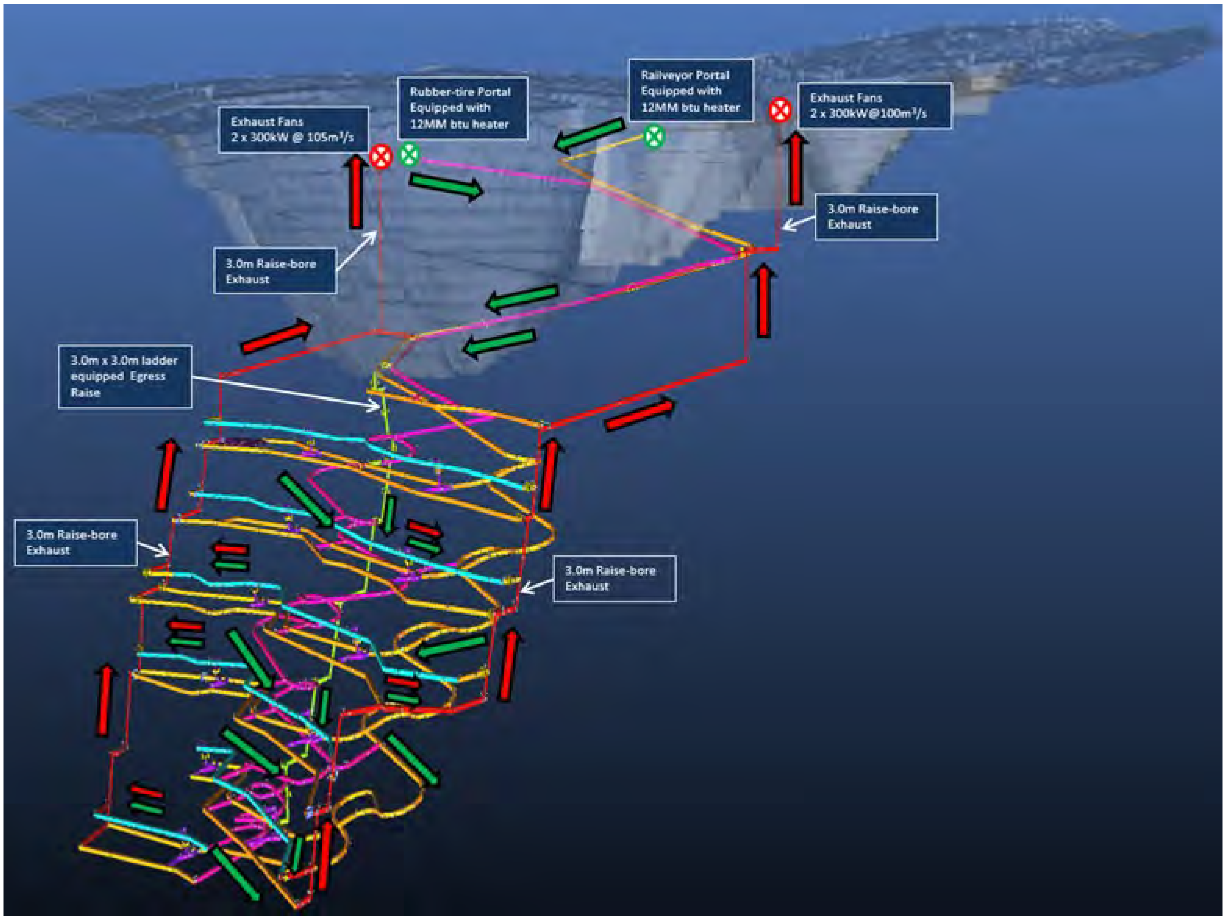
According to the certified CANMET-MSHA approved diesel engines dilution rate and based on the projected equipment list, the maximum fresh air demand at Troilus is estimated at 205m^3/s.

The final Troilus ventilation system is designed as a pull-type exhaust system. Exhaust raises will be located at the north and south extents of the orebody and will be raise bored to a diameter of 3.0m. All air flow will be managed by 2 sets of 2 x 300kW parallel fan installations, with a set located at the top of each of the dedicated exhaust raises. These surface fans will be equipped with variable speed drives.

The exhaust raise system will step down by level through the orebody with regulators equipped at each breakthrough to control the required quantity of air for each operating level. Regulator control is proposed to minimize the requirement for doors and other obstructive measures of ventilation control.

Figure 16-49 illustrates the primary intake and exhaust paths of the of the underground ventilation flows. Key infrastructure installations are also identified.

Figure 16-49: Ventilation Schematic



The fresh air intake system will comprise two declines, one dedicated for the Rail-Veyor equipment and the other for rubber-tired equipment. The portal of each decline will be equipped with a 12MM BTU propane mine-air heater system to supply heated air to the underground workplaces during the winter months. During initial decline development, smaller 2MM btu mine air heaters may be utilized at the portals. Both the Rail-Veyor and diesel equipment will therefore operate in fresh air. Each level will pull the required fresh air from the decline, directing the flow along the working level, and exhausting to the regulator equipped internal exhaust raises.

During decline development and for stope and level development and tramming, ducted auxiliary ventilation systems will be utilized. These systems will be equipped with either 75kW or 100kW fans, depending upon the development purpose and air flow requirements of the diesel equipment being used.

An internal fresh air raise system will be located mid-orebody along the main access decline. This series of small diameter Alimak or drop-type raises will function as "leapfrog" raises to support decline development where appropriate to lessen the requirement for auxiliary ducting. During the production phase, this raise system will serve as an egress from the mine in addition to the twin declines.

16.4.7 Dewatering

Ramp Development

Two ramp declines will be developed – one for rubber tired equipment and one for the Rail-Veyor.

The nominal water inflow is 6 l/s (22 m^3/hr) per ramp.

Four portable pump stations, each equipped with a tank, two 50 HP pumps (1 duty and 1 standby) and a 3 HP agitator to maintain solids in suspension, will pump dirty water. They will be installed down each ramp decline from the surface ramp portal to the first main pump station. Each pump will be capable of delivering 40 m^3/hr at 120 m of head.

An additional portable pump box will be installed down each ramp decline to allow ramp development to progress before the first main pump station is operational. After the main pumping station is operational, the pump boxes will be relocated below as the ramp advances.

Piping in each ramp will consist of one 4" Schedule 40 carbon steel pipe with grooved ends and Victaulic Style 77 couplings.

After the 5050 level pump station is operational, submersible pumps will be installed in the water collection sumps and water will be sent to the 5050L sump.

Production

No crown pillar will be maintained between the open pit and underground mine. As the mining will commence at the open pit/underground interface the open pit will act as a funnel directing surface flows directly into the mine throughout the mine production life. For inflow estimation it was assumed that water diversion ditches are created around the perimeter of the open pit. Rainfall on the footwall benches is assumed to be captured within the open pit and pumped to the surface pond. Freshet and rainfall on the hangingwall benches is assumed to flow into the underground mine.

Rigorous water estimation is yet to be completed however the current estimated water inflows are as follows:

- Ground Water = 120 l/s (432 m^3/hr, 10,368 m^3/day)
- Production = 10 l/s (36 m^3/hr, 864 m^3/day)
- Summer Precipitation = 8.4 l/s (30 m^3/hr, 726 m^3/day)
- Spring Freshet = 25.6 l/s (92 m^3/hr, 2,212 m^3/day)
- Potential Summer Storm = 596 i/s (2,145 m^3/hr, 51,480 m^3/day)

The water inflows were grouped together as follows:

- Minimum = Ground Water + Production
- Maximum = Ground Water + Production + Summer Precipitation + Spring Freshet
- Peak = Ground Water + Production + Summer Precipitation + Spring Freshet + Potential Summer Storm

Four main pump stations will pump clean water from 5050 level, 4850 level, 4650 level and 4500 level. Dewatering drifts will be mined below each S&MB drawpoint level. Drain holes of 152mm diameter will be drilled from the stope drawpoint access to the dewatering drift below. In the dewatering drift captured water will flow towards the pump stations located at the central access ramp. Mine water will be collected in main settling (dirty) sumps that overflow into clean sumps.

The water inflows, number of duty pumps, pump flows and pump duty cycles for each main pump station are summarized in Table 16-20.

Table 16-20: Dewatering Arrangement

		Minimum	Maximum	Peak
5050	Inflow (m^3/hr)	468	590	2,735
	# of Duty Pumps	2	3	4
	Pump Flow (m^3/hr)	700	1,050	1,400
	Pump Duty Cycle	67%	56%	100%
4850	Inflow (m^3/hr)	468	590	2,735
	# of Duty Pumps	2	3	4
	Pump Flow (m^3/hr)	700	1,050	1,400
	Pump Duty Cycle	67%	56%	100%
4650	Inflow (m^3/hr)	351	443	2,051
	# of Duty Pumps	2	2	3
	Pump Flow (m^3/hr)	700	700	1,050
	Pump Duty Cycle	50%	63%	100%
4500	Inflow (m^3/hr)	263	332	1,538
	# of Duty Pumps	2	2	3
	Pump Flow (m^3/hr)	700	700	1,050
	Pump Duty Cycle	38%	47%	100%

The 5050 level pump station will consist of four 8-stage 700 HP Schlumberger REDA HPS horizontal pumps. Each pump will be capable of delivering 350 m^3/hr at 375 m of head.

The 4850 level pump station will consist of four 4-stage 400 HP Schlumberger REDA HPS horizontal pumps. Each pump will be capable of delivering 350 m^3/hr at 220 m of head. The pump station will report to the pump station at 5050 level via boreholes.

The 4650 level pump station will consist of three 4-stage 400 HP Schlumberger REDA HPS horizontal pumps. Each pump will be capable of delivering 350 m^3/hr at 220 m of head. The pump station will report to the pump station at 4850 level via a borehole. The 4650 level water inflows are assumed to be 75% of the 4850 level water inflows. The 4500 level water inflows are assumed to be 75% of the 4650 level water inflows.

During peak water inflow periods, any excess water will be diverted from the sumps and stored in a safe location. The water will then be pumped back to the sumps during off-peak periods.

The 4500 level pump station will consist of three 4-stage 350 HP Schlumberger REDA HPS horizontal pumps. Each pump will be capable of delivering 350 m^3/hr at 160 m of head. The pump station will report to the pump station at 4650 level via a borehole.

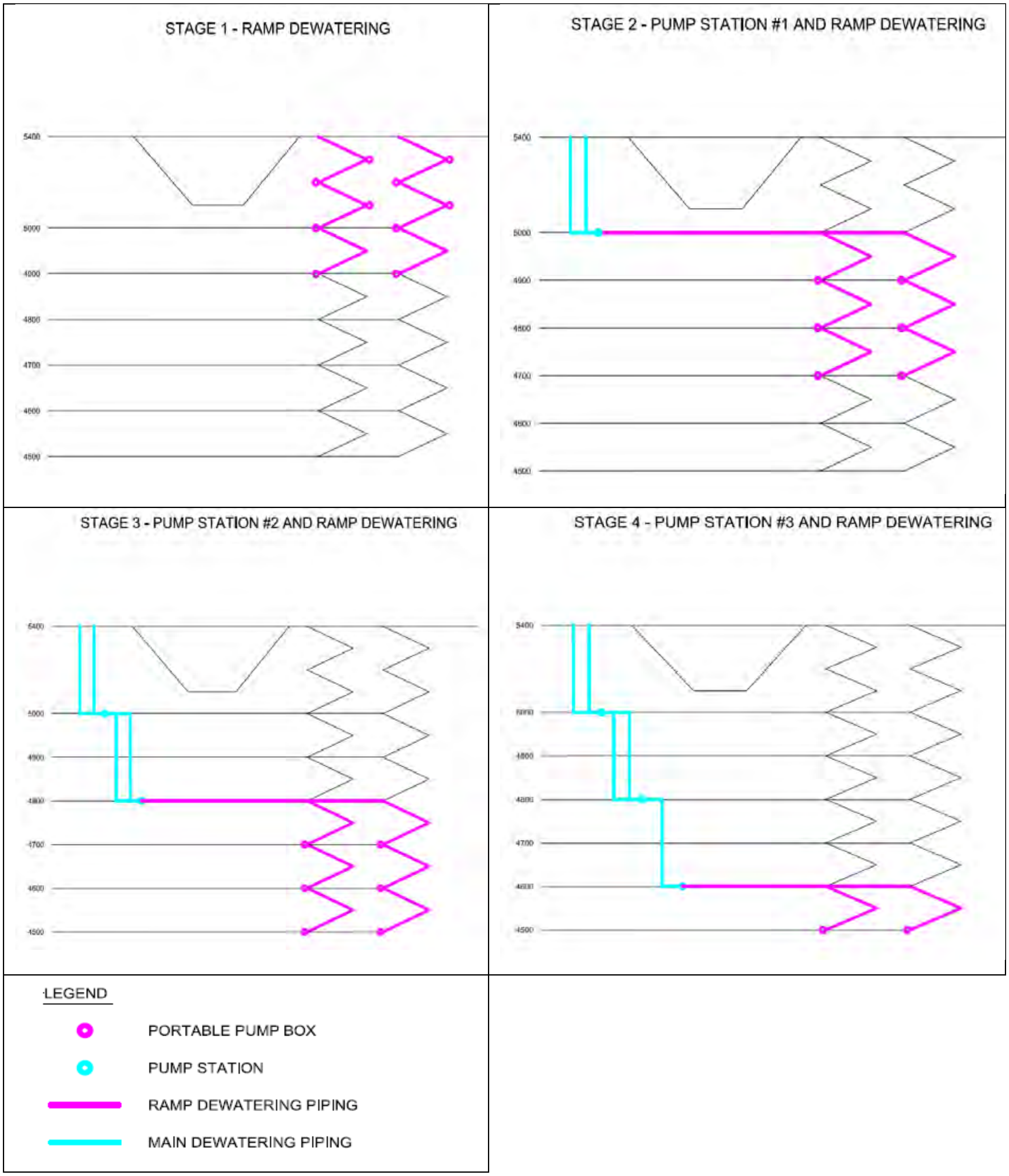
Two 20” boreholes will be drilled from surface to 5050 level and from 5050 level to 4850 level. One 20” borehole will be drilled from 4850 level to 4650 level and from 4650 level to 4500 level.

Piping from surface to 4850 level will consist of two 16” Schedule 40 carbon steel pipes with Class 300 flanges on the levels and with welded plain ends in the boreholes. Each pipeline will convey up to 700 m^3/hr of water. Two pumps will be connected to each pipeline. The duty pumps will be rotated at every pumping cycle.

Piping from 4850 level to 4500 level will consist of one 16” Schedule 40 carbon steel pipe with Class 300 flanges on the levels and with welded plain ends in the boreholes. The pipeline will convey up to 1,050 m^3/hr of water. Three pumps will be connected to the pipeline. The duty pumps will be rotated at every pumping cycle.

Sequential dewatering schematics are shown in Figure 16-50.

Figure 16-50: Sequential Dewatering Schematics



16.4.8 Power Distribution

The mine electrical infrastructure includes provisions to support ramp development, the dewatering pump stations, production activities and the Rail-Veyor material handling system. The underground complex will be supplied via four 13.8 kV feeders, two feeders for production and dewatering loads, and two feeders for the Rail-Veyor material handling system. Total capacity per feeder is approximately 12 MVA. The overall electrical system design gives priority to mine dewatering and the Rail-Veyor material handling system as these are the most critical loads. The surface feeder switchgear and breakers are assumed to be existing.

Ramp Development

Two ramp declines will be developed concurrently – one for rubber-tired equipment and one for the Rail-Veyor system. Electrical will provide two "development" Mine Power Centres (MPC) to supply jumbos and vent booster fans in each ramp. These MPC's will be "leap-frogged" to permit uninterrupted development activities. As ramp cross connects are developed and temporary dewatering pump stations are established an additional MPC will be left behind temporarily. Where MPC's are planned for the Rail-Veyor drive system, permanent electrical rooms will be established in the cross connects. Permanent power and communications cables, supported by a messenger network will be installed as the ramps progress, and terminated where required. Temporary pump station MPC's will be reused for lower ramp development once the first phase of dewatering is established.

Production and Support

The mining plan will only require electrical services on one active production level at a time. MPC's will be provided to support stoping activities as well as material handling, secondary breakage, ventilation, etc. on the working level, and for Rail-Veyor loading on the associated material handling Rail-Veyor spur drift. As a subsequent mining level is established, this will also be serviced electrically in readiness for production.

When mining activities are completed on the first production level, all major electrical infrastructure will be relocated for reuse on the third production level that will be developed. This sequence will repeat for each level until the end of production.

Dewatering

Four main pump stations will be established: 5050 level, 4850 level, 4650 level and 4500 level.

The 5050 level pump station will consist of four 700 HP horizontal pumps. These pumps will have 4.16 kV drive motors. Two 4.16 kV MPC's will be installed to power the pumps and an additional utilities MPC will be provided for the auxiliary loads.

The 4850 level pump station will have four 400 HP pumps and the 4650 level pump station will have three 400 HP pumps. Drive motors will be 4.16 kV and each station will be equipped with two 4.16 kV MPC's to power the pumps and an additional utilities MPC for the auxiliary loads.

The 4500 level pump station will consist of three 350 HP pumps. These pumps will have 600 V drive motors and the pump station will be equipped with a 600 V MPC to supply all associated loads.

Material Handling

The transport of materials from depth to surface will be accomplished using Rail-Veyor technology. The current proposed layout will require 16 MPC's to power 256 drive stations, each of which is rated 200 HP. Total running load is estimated at less than 2000 HP but due to the distributed power input (drive stations) a large number of MPC's and the associated electrical infrastructure is required. As a result, the Rail-Veyor system will be provided with dedicated 13.8 kV feeders to better manage the power distribution system.

16.4.9 Safety

Underground Communications

The proposed technology for communications and automation will be through a primary fiber optic trunk network. Fiber interface panels will be located at key locations to provide interconnectivity to associated users. Primary user networks will be voice and data; Rail-Veyor automation; security, access control, and personnel/vehicle tracking; geotechnical monitoring; process automation; and engineering and maintenance.

In the main access ramp, a sub-network employing a Plexus PowerNet system with wireless access points for full LTE coverage and provisions for additional devices such as ventilation / air quality monitoring, and other equipment as may be required will be installed. This will permit mobile telephone or tablet communications and data transfer similar to nominal surface services. In the Rail-Veyor ramp a similar system will be used, where the Rail-Veyor drive stations are utilizing the network, and wireless access points, maintenance laptops etc. will be plugged in as required. The full Rail-Veyor ramp will not normally have 100% wireless coverage, as this area is normally unoccupied for safety reasons.

The geotechnical monitoring and the engineering and maintenance sub-networks will be separate to provide reliability without interference from other users. The process automation and monitoring sub-network is expected to require a significant amount of bandwidth depending on the final design and the location of operators. Video monitoring for remote operation may also be incorporated into this sub-network.

Security, access control, and personnel and vehicle tracking sub-network will provide the necessary interconnectivity for tracking personnel and vehicles, including the necessary information for mine rescue purposes.

The final communications design will require further optimization based on final mine design and communications requirements. A very conservative approach has been taken at this preliminary stage to ensure overall system capability.

A hardwired telephone system is also proposed as an emergency back up to the electronics based fiber-optic network. The telephone system will be in addition to the wireless voice network and will only be installed in refuge stations and central mine rescue muster stations.

Fire Prevention

All diesel equipment (light vehicles and heavy-duty mobile equipment) will be equipped with automatic fire suppression systems and hand-held fire extinguishers. Hand-held fire extinguishers will be located throughout the mine at refuelling bays, workshops, explosive and detonator magazines, refuge bays, and lunchrooms. Refueling bays, workshops, explosive and detonator magazines will be equipped with automatic deluge systems.

A mine-wide stench gas system will be installed at the fresh air intakes to alert underground workers in the event of an emergency.

Mine Rescue

A mine rescue team will consist of members selected and trained from the owners' workforce and at least two teams would be available on rotation for rescue efforts. Surface and underground training facilities will be necessary for ongoing employee training and refresher training programs. Dedicated mine rescue equipment including a rescue vehicle and all supporting testing and maintenance equipment for mine rescue purposes will be available and specific underground mine rescue equipment would include self-contained breathing apparatus.

Refuge Stations

A lunchroom/refuge station will be located on each S&MB mucking level. Portable refuge chambers accommodating from 12 to 16 people are planned for the initial development period and for subsequent isolated areas. The refuge stations will be equipped with independent compressed air supply piped from surface, CO and CO_2 scrubbers, medical oxygen cylinders, oxygen candle, air conditioning, first aid kit, radio, telephone, and a toilet. They would provide protection for at least 36 hours as a self-contained unit. Access will be through an air lock door and they will be equipped sealant to hermetically seal the doors in order to prevent the entry of harmful gases.

Emergency Egress

Independent emergency egress to surface is provided by the separate main access and Rail-Veyor ramps. In order to provide second egress from each of the sub-levels a raise with manway ladder will be mined at both the north and south extents between each S&MB drawpoint level and the upper drilling/Rail-Veyor level below. In the event of fire underground preventing safe retreat through the workforce will report to the refuge stations.

Dust Control

Spray nozzles operated by the mobile equipment drivers will be installed at all material loading points for dust control.

16.4.10 Underground Mine Development and Production Schedules

The natural progression of stoping for both the SLC and S&MB mining methods is following a top-down path though the orebody. This stoping philosophy matches well with the Rail-Veyor materials handling system which can be progressively and easily extended to depth without interfering with stoping and materials handling activities above. As such, pre-production development requirements of this arrangement are less than that required for a more conventional approach to materials handling than a system that requires ore passes and underground crushing as part of the system.

Development Schedule

The development lengths of the various components of the mine design summarised in Table 16-21 were quantified directly from the centerline design strings. These quantities were factored upwards as shown in Table 16-16 to recognise additional development requirements not specifically designed, such as electrical transformer stations, material storage bays, and local pump station requirements.

Table 16-21: Mine Development Factors by Development Type

	Dev. Factors	Drift Width	Drift Height
Vehicle Ramp	1.02	5.0	5.5
Materials Handling - Rail-Veyor Ramp	1.00	6.0	5.0
Materials Handling - Rail-Veyor - Other	1.00	5.0	5.0
Level Access	1.80	5.0	5.5
Level	1.00	5.0	5.5
Ventilation	1.02	4.5	5.0
Dewatering	1.00	5.0	5.0
Orepass/Maintenance/Ancillary	1.00	5.0	4.5

The mine development rates used to schedule the quarterly advance of each development crew vary based on the number of headings available to each crew at any particular scheduling period and whether development is to be undertaken by mining contractor or owner crews. The quarterly development rates used in this study are shown in Table 16-22. A total of three contractor development crews were required to complete pre-production development. These contractor crews were sequentially phased out and each replaced by owner-operator development crews. The initial mine development rate for a new development crew was reduced to 75% of the values shown below for the first 3 months of mine development by that crew.

Table 16-22: Quarterly Development Rates

	Heading Availability		
Development Crew	Single	Double	Multiple
Contractor	450	630	780
Owner	390	540	690

Mine development commenced in Year 6 with two contract crews, with each crew developing both the vehicle ramp and the Rail-Veyor ramp. One crew began on surface at the portal and box cut locations, and the other crew began development downwards from the temporary mine access portal located within the confines of the open pit, located approximately 175 vertical meters below the portal/box cut elevations. A third contractor development crew was introduced into the mine schedule at the end of Year 8, once the first leg of the two exhaust raise systems were excavated to surface and equipped with their primary ventilation fans.

Once the third development crew commenced mine development, the focus was to develop the mine dewatering infrastructure in the upper portion of the mine required prior to commencing SLC stoping below the open pit floor. This development formed the critical path to the start of mill feed production. As a consequence of the dewatering critical path, underground stope production commenced during development Year 10/Quarter 1, whilst production from the S&MB slot stopes commenced during development Year 11/Quarter 2. A summary of the annual mine development schedule is shown in Table 16-23.

Production Schedule

Previous trade off study work indicated that a production rate of around 9,000 tpd of mill feed would be achievable.

Mill feed production is planned to begin at a central location along the SLC stoping block located directly below the open pit floor. SLC stoping will progress to the extremities of the level and incorporate mining of the SLC levels adjacent to the end walls of the open pit. SLC stoping on the level below will commence directly below the previous start location once SLC stoping on the upper level has progressed sufficient to expose the lower SLC level to the cave waste. In this way a considerable number of SLC draw points will be active at any one time and will be able to sustain a relatively high production rate. SLC stope production was scheduled at a maximum rate of 280 tonnes per day per available drawpoint. SLC stoping was carried out over a 3 ½ year period at an average steady state production rate of approximately 6,000 tpd. The maximum SLC production rate on a quarterly basis was 7,400 tpd.

The productive capacity of the S&MB stopes is largely dependent on the LHD mucking rate from the stope drawpoint to the closest rock bin on the level. The relatively short tramming distance to one of the two rock bins on each mucking level result in a high calculated stope production rate. The calculated stope production rate for 1, 2, and 3 loaders mucking from the herringbone drawpoint arrangement under the two trough undercuts available from each S&MB stope result in calculated production rates of 2,700 tpd, 5,400 tpd, and 7,300 tpd, respectively. As such, only two active S&MB stopes would need to be in production at any one time to provide the full 9,000 tpd of underground mill feed. However, in order to provide a higher degree of flexibility and reliability of production, mine development was scheduled such that mill feed production was available from four to six S&MB stopes at any point in time. As a result, the maximum production rate from a slot stope (with a single set of herringbone draw points) was scheduled at no greater than 2,700 tpd, whilst the production rate following a mass blast (with dual herringbone drawpoint arrangements available) was scheduled at a maximum of 3,300 tpd. These represent maximum scheduled production rates - in many cases the scheduled production rate per stope was in the order of 500 tpd less than the values stated above.

The S&MB stoping arrangement as designed is capable of producing from up to six stopes at any one time: S&MB stoping commences at a central location on a level and progresses to the extremities of the level in both directions, thereby creating four separate working places on a level (i.e. mucking from four trough undercuts). Further, once the mass blast portion of an S&MB stope has been mucked out and replaced by waste material from above, the mass blast of the S&MB stope on the level below can be fired, having already blasted and mucked the slot stope empty in readiness for the mass blast firing. In this way, two and sometimes three S&MB levels can be in production at any one time – with multiple working places on each level. The added benefit of multi-level S&MB stoping is that the maximum duty required of each of the grizzly/rockbreaker/rockbin/Rail-Veyor feeder arrangements is in the order of only 2,500 to 3,000 tpd, which would be readily achievable with this arrangement. Mill feed production from the S&MB stopes start approximately one year after the start of SLC production and continue for 12 ½ years.

A summary of the annual mine production schedule is shown in Table 16-24. End of year plans showing completed mine development and stoping activities are provided in Section 16-6.

Table 16-23: Annual Summary – Development and Production Schedule

	Total	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13
SLC Stope Mill Feed (t)	**4,827,033 t**				2,031,569	1,908,698	841,996	44,770	
SLC Dev Mill Feed (t)	**472,242 t**			105,851	159,105	139,955	67,330		
SLC Dev Mill Feed (m)	**7,174 m**			1,600	2,420	2,131	1,023		
Total Mill Feed (t)	**5,299,275 t**			105,851	2,190,674	2,048,653	909,326	44,770	
Eq Au g/t	**1.38**			1.39	1.33	1.40	1.42	1.41	
Au g/t	**1.18**			1.20	1.14	1.20	1.22	1.22	
Cu %	**0.13**			0.13	0.12	0.13	0.13	0.12	
Ag g/t	**1.33**			1.63	1.43	1.26	1.24	1.15	
MB Up Dev Mill Feed (m)	**7,217 m**					289	613	634	554
MB Up Dev Waste (m)	**3,802 m**		60	360	255	96	469	265	344
Stp Up Dev Mill Feed (m)	**6,589 m**					382	636	630	683
Stp Up Dev Waste (m)	**1,898 m**		30	195	120	107	209	136	164
MB Body Mill Feed (t)	**22,496,706 t**					459,276	1,199,316	2,002,928	2,125,570
MB Undercut Mill Feed (t)	**2,169,897 t**					90,635	122,162	188,421	170,848
MB Drwpnt Mill Feed (m)	**4,119 m**					328	241	257	448
Stp Body Mill Feed (t)	**7,851,886 t**					306,544	667,579	667,987	549,322
Stp Undercut Mill Feed (t)	**1,790,099 t**					64,174	160,082	162,945	138,251
Stp Drwpnt Mill Feed (m)	**20,295 m**				178	1,283	1,744	1,967	2,232
Stp Drwpnt Waste (m)	**2,361 m**				461	139	68	353	361
Total Dev Waste (m)	**8,061 m**		90	555	836	342	746	754	869
Total Dev Mill Feed (m)	**38,220 m**				178	2,282	3,234	3,488	3,917
Total Mill Feed (t)	**37,016,552 t**				11,904	1,105,916	2,375,674	3,240,237	3,284,999
Eq Au g/t	**1.34**				1.36	1.61	1.44	1.33	1.38
Au g/t	**1.18**				1.17	1.39	1.23	1.13	1.17
Cu %	**0.10**				0.12	0.14	0.14	0.13	0.14
Ag g/t	**0.79**				1.72	1.99	1.85	1.72	1.46
Total Dev Waste (m)	**52,575 m**	4,680	5,650	9,284	7,362	2,987	3,143	3,882	3,483
Total Dev Mill Feed (m)	**45,394 m**			1,600	2,598	4,413	4,257	3,488	3,917
Total Mill Feed (t)	**42,315,827 t**			105,851	2,202,578	3,154,569	3,285,001	3,285,007	3,284,999
Eq Au g/t	**1.35**			1.39	1.33	1.47	1.43	1.33	1.38
Au g/t	**1.18**			1.20	1.14	1.27	1.23	1.13	1.17
Cu %	**0.11**			0.13	0.12	0.13	0.13	0.13	0.14
Ag g/t	**0.86**			1.63	1.43	1.51	1.68	1.71	1.46
TPD				290	6,034	8,643	9,000	9,000	9,000

Table 16-24: Annual Summary – Development and Production Schedule (cont'd)

	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22
SLC Stope Mill Feed (t)									
SLC Dev Mill Feed (t)									
SLC Dev Mill Feed (m)									
Total Mill Feed (t)									
Eq Au g/t									
Au g/t									
Cu %									
Ag g/t									
MB Up Dev Mill Feed(m)	802	517	650	626	749	626	590	525	42
MB Up Dev Waste (m)	361	454	210	195	310	159	110	154	
Stp Up Dev Mill Feed (m)	478	608	488	601	659	585	442	397	
Stp Up Dev Waste (m)	213	196	162	104	63	108	50	41	
MB Body Mill Feed (t)	1,670,461	1,871,130	2,018,380	1,950,253	2,088,067	1,978,103	1,974,986	2,405,118	753,118
MB Undercut Mill Feed (t)	162,420	298,922	189,635	249,786	111,192	173,893	249,819	162,164	
MB Drwpnt Mill Feed (m)	441	491	309	514	343	346	249	152	
Stp Body Mill Feed (t)	985,985	735,210	647,311	639,335	687,470	727,698	764,380	473,065	
Stp Undercut Mill Feed (t)	232,990	127,591	167,386	175,209	180,792	167,445	129,055	84,179	
Stp Drwpnt Mill Feed (m)	1,553	2,005	2,320	2,135	1,337	1,821	1,072	648	
Stp Drwpnt Waste (m)	54	333	239	193	75	20	51	14	
Total Dev Waste (m)	628	983	611	492	448	287	211	209	
Total Dev Mill Feed (m)	3,274	3,621	3,767	3,876	3,088	3,378	2,353	1,722	42
Total Mill Feed (t)	3,285,000	3,285,000	3,285,000	3,285,000	3,285,000	3,285,000	3,285,001	3,246,700	756,121
Eq Au g/t	1.37	1.14	1.16	1.24	1.29	1.44	1.35	1.55	1.42
Au g/t	1.19	0.99	1.00	1.10	1.15	1.30	1.23	1.43	1.31
Cu %	0.11	0.10	0.10	0.09	0.09	0.09	0.08	0.08	0.07
Ag g/t	1.27	0.79	0.75	0.23	0.39	0.15	0.10	0.07	0.01
Total Dev Waste (m)	4,042	3,779	2,636	492	448	287	211	209	
Total Dev Mill Feed (m)	3,274	3,621	3,767	3,876	3,088	3,378	2,353	1,722	42
Total Mill Feed (t)	3,285,000	3,285,000	3,285,000	3,285,000	3,285,000	3,285,000	3,285,001	3,246,700	756,121
Eq Au g/t	1.37	1.14	1.16	1.24	1.29	1.44	1.35	1.55	1.42
Au g/t	1.19	0.99	1.00	1.10	1.15	1.30	1.23	1.43	1.31
Cu %	0.11	0.10	0.10	0.09	0.09	0.09	0.08	0.08	0.07
Ag g/t	1.27	0.79	0.75	0.23	0.39	0.15	0.10	0.07	0.01
TPD	9,000	9,000	9,000	9,000	9,000	9,000	9,000	8,895	2,072

16.4.11 Mine Costing Methodology

Costs for the underground mine plan were estimated quarterly throughout the life of mine from first principles. A series of unit cost models was adapted to reflect the direct activities at the mine. Each of the models was developed reflecting the mine design criteria and other general engineering estimates of performance. The mine was assumed to work on two 12-hour shifts per day, 365 days per year. All costs were modelled as end 1st Quarter 2020 Canadian Dollars.

It is planned that all capital development during the project preparation phase to Q2 Year 9 will be undertaken by contractors. Owner crews will commence operating development during the project preparation phase. All activities will be undertaken by owners' crews after Q2 Year 9 apart from raising and delineation drilling which will be completed by a contractor. Contractor unit rates included equipment rentals, 15% cost mark-up on labour, consumables, and equipment plus a 40% allowance for contractor indirect charges. The development cost models include ground support assumptions provided by the geotechnical study. The unit rates were applied to the scheduled quantities in order to estimate the direct costs.

Additional models were designed to reflect overhead-type activities at the mine:

- mobile equipment leasing costs
- rail-Veyor operations (including operating grizzly tips and Rail-Veyor loading, comprising operating and maintenance labour, maintenance supplies and equipment operation)
- mine services and fixed plant (including labour, supplies, and equipment for secondary breakage in stopes, construction, materials transport, road maintenance, dewatering and sanitation); diesel maintenance labour costs are also included
- owners mine management and technical (including mine supervision, mine technical and safety staff)
- mine air heating
- mine power (developed from aggregation of mine loads and estimated usage)

Overheads were estimated on a quarterly basis and applied as a fixed daily cost. The overheads for each period were split between operating and capital development estimates in the ratio of the respective direct costs.

Consumable and material unit pricing for underground mining activities, were applied using data from other recent AGP mining projects. Where necessary, costs were escalated at the rate of 3% per year from the date of information to reflect end 1st Quarter 2020 costs.

Labour rates were based on an industry survey of Quebec underground mine pay rates.

The models were also used to track labour and equipment hours to identify quarterly requirements in each labour category and equipment type.

A summary of the unit costs derived during the modelling process is shown in Table 16-25.

Table 16-25: Summary of Underground Unit and Overhead Costs

		CAN$	
	Unit	Owner	Contractor
Ramp - 5.0m wide x 5.5m high	m	2,398	3,843
Level Waste Drift - 5.0m wide x 5.5m high	m	2,295	3,797
Vent/Other Drift - 4.5m wide x 5.0m high	m	1,798	2,967
Materials Handling - 5.0m wide x 5.0m high	m	2,182	3,604
Rail-Veyor Drift - 6.0m wide x 5.0m high	m	2,392	3,956
Dewatering Drift - 5.0m wide x 5.0m high	m	1,836	3,032
Workshop/Pumps - 5.0m wide x 4.5m high	m	2,896	4,728
Waste Stope X/C - 4.6m wide x 4.6m high	m	1,933	3,204
S&MB Drawpoint Drift - 4.6m wide x 4.6m high	m	4,053	
S&MB Ore Drift - 4.6m wide x 4.6m high	m	2,715	
SLC Ore Drift - 6.0m wide x 4.8m high	m	2,623	
Conventional Alimak Raise - 1.8m wide x 1.8m high with Ladder	m		3,505
Longhole Raise 3.0 x 3.0m Drill & Blast	m	1,350	
Raise bore (3.5m dia)	m		9,161
Raise bore (4.0m dia)	m		10,292
Drainhole	m	84.63	
SLC Drilling And Blasting	SLC t	4.64	
S&MB Trough Drilling And Blasting	Trough t	2.48	
Slot & Mass Blast Drilling & Blasting	Stope & Mass Blast t	2.84	
S&MB Scoop Mucking - LHD From Stope To Level Tip	t	1.76	
SLC Scoop Mucking - LHD From Stope To Level Tip	t	1.93	
Waste Trucking to Surface or Open pit			
50m Vertical Haul	t	3.99	7.90
150m Vertical Haul	t	6.35	13.46
250m Vertical Haul	t	8.70	19.01
350m Vertical Haul	t	11.06	24.57
Millfeed/Waste Trucking to Rail-Veyor			
50m Vertical Haul		3.46	7.90
Contract Diamond Drilling (Delineation)	Stope t		0.30
Mobile Equipment Leasing Charges [1]	Day	15,386	
Rock Tips & Rail-Veyor [1]	Day	27,014	
Mine Services, Fixed Plant & Mobile Equipment Maintenance Labour [1]	Day	37,467	
Owners Mine Supervision & Technical [1]	Day	25,750	
Mine Air Heating (Propane) [1]	Day	4,645	
POWER [1]	Day	5,938	

Note: Overheads provided indicate scale of expense. Modelled estimate overheads vary in each estimate period depending on mine activity. Mobile equipment leasing charges were also estimated by quarterly calculation.

16.4.12 Labour

Labour force plans were developed to support the life of mine plan and the activities scheduled to meet production objectives. The labour force tables provided reflect the underground workforce required to support development, production, and associated activities.

Hourly paid employees will workday and night shifts, each of 12 hours, on a rotating 7 days on and 7 days off schedule for 365 mine-operating days per year. Each hourly paid employee will work 2,147 h/a after allowances for overtime, vacation, absenteeism, and sickness.

To provide for continuous operations there will be two employees per position.

Senior manager, supervisor, and technical staff will work 2,000 h/a. At operational level there are two employees per position. For more senior and technical positions there will be one employee per position with work based on four 10 hour working days per week.

Job categories will be manned one or two shifts per day basis depending on the position.

Table 16-26 shows the estimate of personnel employed by the owner for selected periods during the production phases of the life of mine.

Table 16-26: Owners Employed Labour

	Y7 Q1	Y9 Q1	Y11 Q1	Y13 Q1	Y15 Q1	Y17 Q1	Y19 Q1	Y21 Q1
Hourly Paid								
Rail-Veyor Operator	-	4	4	4	4	4	4	4
Longhole Drilling	-	8	9	6	8	5	6	6
Development Miner	-	18	19	18	15	14	11	8
Secondary Breakage	-	4	4	4	4	4	4	4
Scoop Driver	-	15	20	17	16	15	14	14
Stope Blasting	-	4	12	15	14	16	15	16
Rockbreaker	-	20	20	20	20	20	20	20
Construction	6	7	9	8	8	9	9	8
Materials	-	8	8	8	8	8	8	8
Truck Driver	-	10	10	5	4	4	3	2
Laborer	-	26	33	34	33	33	31	26
Pumps	-	4	4	8	8	8	8	8
Road Maintenance	2	2	2	2	2	2	2	2
Mechanic I	2	4	4	4	4	4	4	4
Mechanic III	-	4	4	4	4	4	4	4
Mechanic III	2	4	4	4	4	4	4	4
Electrician I	2	4	4	4	4	4	4	4
Electrician II	-	8	8	8	8	8	8	8
Electrician III	2	4	4	4	4	4	4	4
Diesel Mech I	1	13	14	13	12	12	11	10
Diesel Mech II	1	13	14	13	12	12	11	10
Diesel Mech !!I	1	13	14	13	12	12	11	10
Total Hourly Paid	**19**	**197**	**224**	**216**	**208**	**206**	**196**	**184**
Staff								
Maintenance Supt	-	1	1	1	1	1	1	1
Maintenance Foreman	1	4	4	4	4	4	4	3

	Y7 Q1	Y9 Q1	Y11 Q1	Y13 Q1	Y15 Q1	Y17 Q1	Y19 Q1	Y21 Q1
Maintenance General Foreman	-	2	2	2	2	2	2	2
Maintenance Planner	-	-	1	1	1	1	1	-
Mine Superintendent	1	1	1	1	1	1	1	1
Mine Captain	-	1	3	3	3	3	3	2
Shift Boss	-	8	16	16	16	16	16	12
Mine Dry/Lamps/Bits	-	2	2	2	2	2	2	2
Secretary/Clerk/Stores	2	4	4	4	4	4	4	3
Safety	1	1	1	1	1	1	1	1
Chief Engineer	1	1	1	1	1	1	1	-
Senior Mine Engineer	1	1	1	1	1	1	1	1
Chief Geologist	1	1	1	1	1	1	1	-
Senior Geologist	1	1	1	1	1	1	1	1
Mine Geologist	1	3	3	3	3	3	3	2
Mine Technician	1	3	3	3	3	3	3	2
Geology Technician/Grade Control	1	2	2	2	2	2	2	2
Mine Engineer	1	4	4	4	4	4	4	3
Surveyor	2	2	2	2	2	2	2	2
Survey Helper	4	4	4	4	4	4	4	4
Portal Attendant	4	4	4	4	4	4	4	4
Ventilation /Samplers/Rockmechanics Asst	-	4	4	4	4	4	4	4
Total Staff	**23**	**54**	**65**	**65**	**65**	**65**	**65**	**52**
Total Employed Labour	**42**	**251**	**289**	**281**	**273**	**271**	**261**	**236**

16.4.13 Equipment

Modelled equipment requirements are based on operational hours. Budget quotations received from potential suppliers were used for the equipment types selected. These quotations were escalated by 3% pa from the date of quotation where necessary to reflect end 1st Quarter 2020 values. Mechanical availability, utilization, and operational life were estimated by AGP for each equipment type and the hourly operating costs were assessed. Equipment rentals and profit elements were added when considering contractor equipment rates.

Leasing terms of 20% down payment followed by a five year payment period at the rate of 1.78% per month were applied against all the owners new and replacement mobile equipment acquisitions. A mid-life rebuild equivalent to 50% of the purchase price was assumed for all items of equipment in order to extend the useful life. The entire rebuild costs were classified as sustaining capital.

The mine will contain a significant diesel fleet, which poses a fire hazard. All vehicles will be fitted with on-board detection and suppression systems and in addition, a mine-wide fire detection system is recommended. The underground diesel workshop will be ventilated directly via a main return raise.

As the activities vary, the equipment fleet requirements change. Table 16-27 shows the owners fleet requirements for example periods in the life of mine plan.

Table 16-27: Owners Equipment Requirements

	Y7 Q1	Y9 Q1	Y11 Q1	Y13 Q1	Y15 Q1	Y17 Q1	Y19 Q1	Y21 Q1
6.7t Scoop	1	2	2	2	2	2	2	2
21t Scoop		2	4	4	4	4	4	4
10t Scoop		3	3	2	2	2	1	1
40 t Diesel Truck		3	3	1	1	1	1	1
2 Boom Development Jumbo		3	3	3	2	2	2	1
Top hammer Longhole Drill		2	2	1	1		1	1
ITH Longhole/ Raise Borer		2	1	3	3	2	2	2
Rockbolter		2	3	2	2	2	2	1
Blockholer		1	1	1	1	1	1	1
High Hang-up Drill		1	1	1	1	1	1	1
Boom Truck		2	2	2	2	2	2	2
Fuel/Lube		1	1	1	1	1	1	1
Shotcrete		2	2	2	2	2	2	2
8 Man Personnel		3	3	3	3	3	3	3
Scissors	1	5	5	5	5	5	4	4
Transmixer		1	1	1	1	1	1	1
Emulsion Loader		3	3	3	3	3	3	3
Grader		1	1	1	1	1	1	1
Mobile Breaker		1	1	1	1	1	1	1
Toyota Runaround	4	7	8	8	8	8	8	7
Mechanics Runaround		2	2	2	2	2	2	2
Rescue/First Aid	1	1	1	1	1	1	1	1
Telehandler		2	2	2	2	2	2	2
Stationary Breaker		4	4	4	4	4	4	4
Water Jet		1	1	1	1	1	1	1
Sanitation	1	1	1	1	1	1	1	1
Continuous Loader		4	4	4	4	4	4	4

16.4.14 Power and Mine Air Heating

A load list was estimated for the underground mine which is summarised in Table 16-28.

Table 16-28: Summary Total Installed Mine Power

	Installed kW
Ventilation	3,275
Grizzley & Rail-Veyor	39,215
Dewatering	5,469
Mobile Compressors	240
Mine Equipment	1,680
Other Loads	631
Total	**50,511**

The load list was examined by period to estimate the power usage rate in the mine plan as shown in Figure 16-51. Total power usage and cost is shown in Figure 16-52.

Figure 16-51: Power Usage Rate

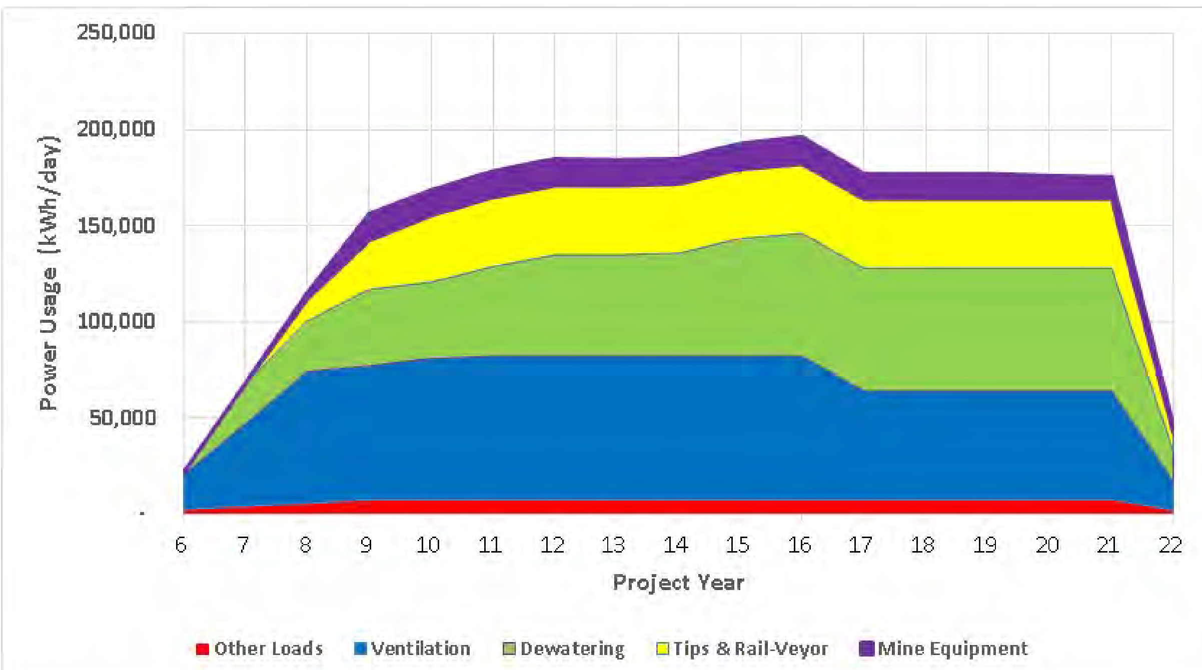
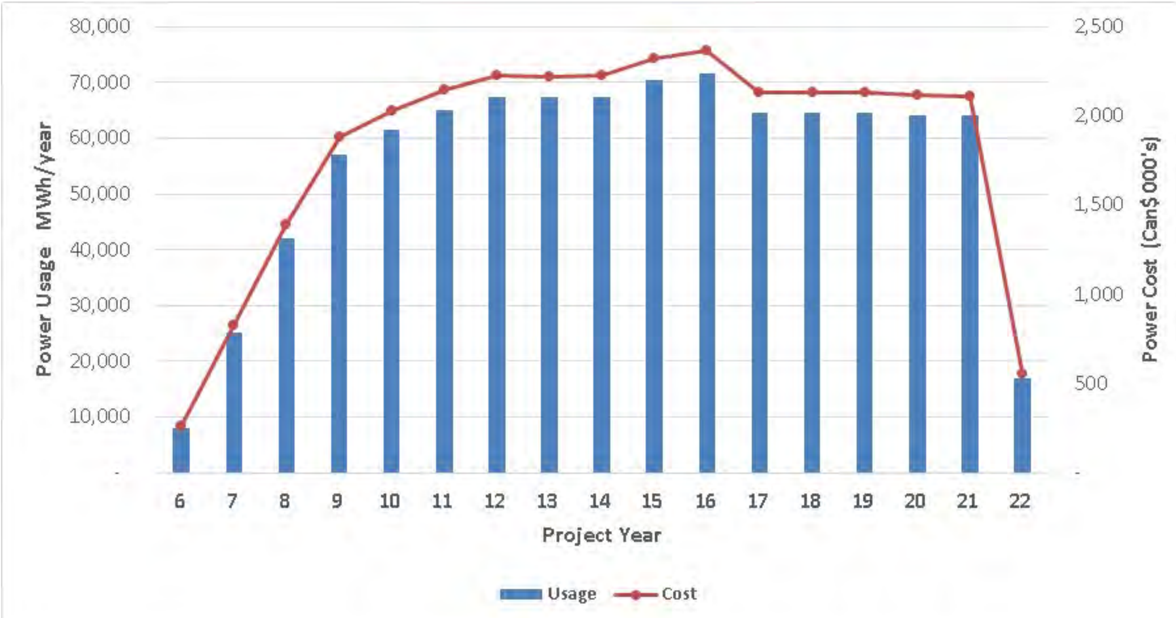


Figure 16-52: Total Power Usage and Cost



Mine air heating costs using propane for maximum ventilation airflow were estimated using local climate data and a minimum mine air temperature of +2^{0}C as shown in Table 16-29. The heating costs were subsequently adjusted for planned quarterly airflow estimates.

Table 16-29: Steady-State Estimated Mine Air Heating Costs

	Q1	Q2	Q3	Q4	Annual
Maximum Airflow					205 m^3/sec
Extreme Maximum Temp Annual Heating Costs	-	-	-	-	-
Average Temp Annual Heating Costs	917,841	41,916	-	424,993	1,384,750
Extreme Minimum Temp Annual Heating Costs	2,368,382	1,011,515	273,852	1,699,911	5,353,659
Likelihood: Average 80%, Extreme Min 10%, Extreme Max 10%					
	971,111	134,684	27,385	509,985	1,643,166

16.5 Combined Production Schedule

The combined mine schedule for open pit and underground mining consists of 192.5 Mt of mill feed grading 0.71 g/t gold, 0.076% copper, and 0.97 g/t silver over a mine life of twenty-two years. Open pit waste tonnage totals 591 Mt and will be placed into waste storage areas. The overall open pit strip ratio is 3.94:1. The mine schedule utilizes the pit phase and underground designs described previously to send a maximum of 12.6 Mtpa of feed to the mill facility.

The current mine life includes one year of pre-stripping followed by twenty-two years of mining. Mill feed is stockpiled during the pre-production year. A peak stockpile capacity of 8.9 Mt was reached near the end of year 6 and was useful to support the inclusion of underground mill feed in year 8.

The timing of open pit mining total tonnes and underground mill feed are displayed in Table 16-30 and Figure 16-53. A maximum descent rate of 6 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 -1 and continued uninterrupted until year 14. This schedule included underground mill feed becoming available in year 8 and continuing until year 22. The mill was run at full capacity until year 13, then was reduced to a lower annual mill throughput based on availability of underground mill feed. Process tonnages and gold grade are shown in Figure 16-54.

Table 16-30: Annual Material Mined by Source (excluding u/g development)

Period	Scheduled Material by Source							Total
	Z87 Pit	J Pit			SW Pit		Underground	
	phase 1 (Mt)	phase 1 (Mt)	phase 2 (Mt)	phase 3 (Mt)	phase 1 (Mt)	phase 2 (Mt)	Mill Feed (Mt)	(Mt)
Y-1	27.2				12.3			39.5
Y1	51.7	0.0			21.3			73.0
Y2	41.5	22.6			8.1			72.3
Y3	31.5	36.9				1.6		70.0
Y4	20.9	10.7	6.8			31.6		70.0
Y5	12.3	0.5	36.6			20.6		70.0
Y6	1.0	18.5	31.7			13.8		65.0
Y7		7.2	29.1	11.2		2.5		50.0
Y8			21.7	34.7			0.1	56.5
Y9			6.3	40.7			2.2	49.2
Y10				43.3			3.2	46.4
Y11				35.4			3.3	38.7
Y12				28.9			3.3	32.2
Y13				17.9			3.3	21.1
Y14				2.5			3.3	5.8
Y15							3.3	3.3
Y16							3.3	3.3
Y17							3.3	3.3
Y18							3.3	3.3
Y19							3.3	3.3
Y20							3.3	3.3
Y21							3.2	3.2
Y22							0.8	0.8
Total	**186.0**	**96.5**	**132.2**	**214.7**	**41.8**	**70.0**	**42.3**	**784**

Figure 16-53: Tonnes Mined by Phase

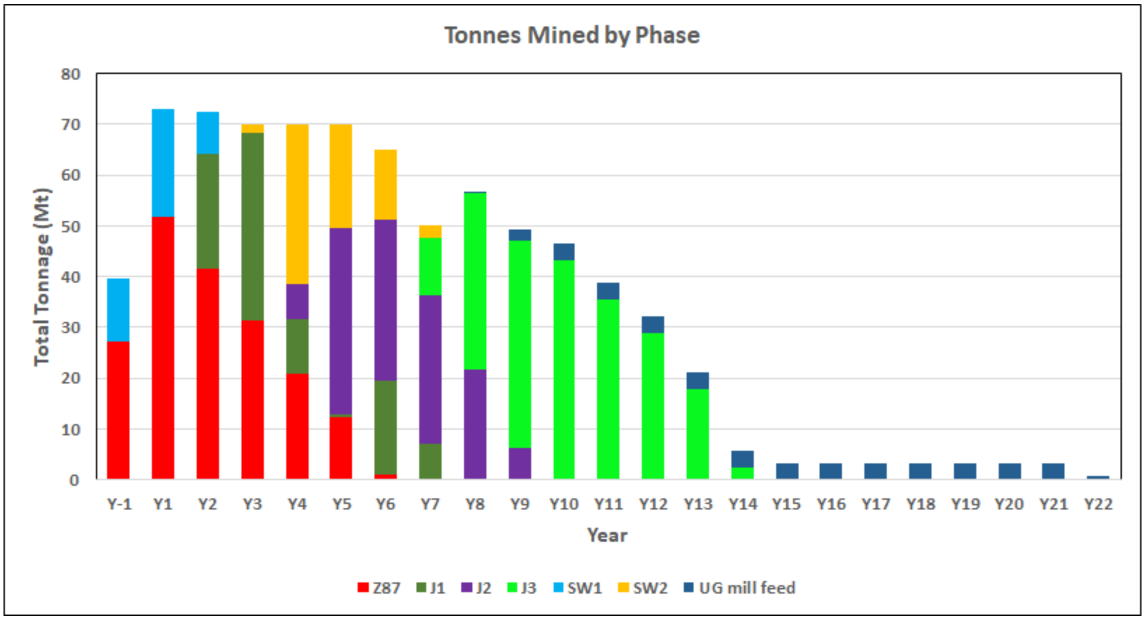
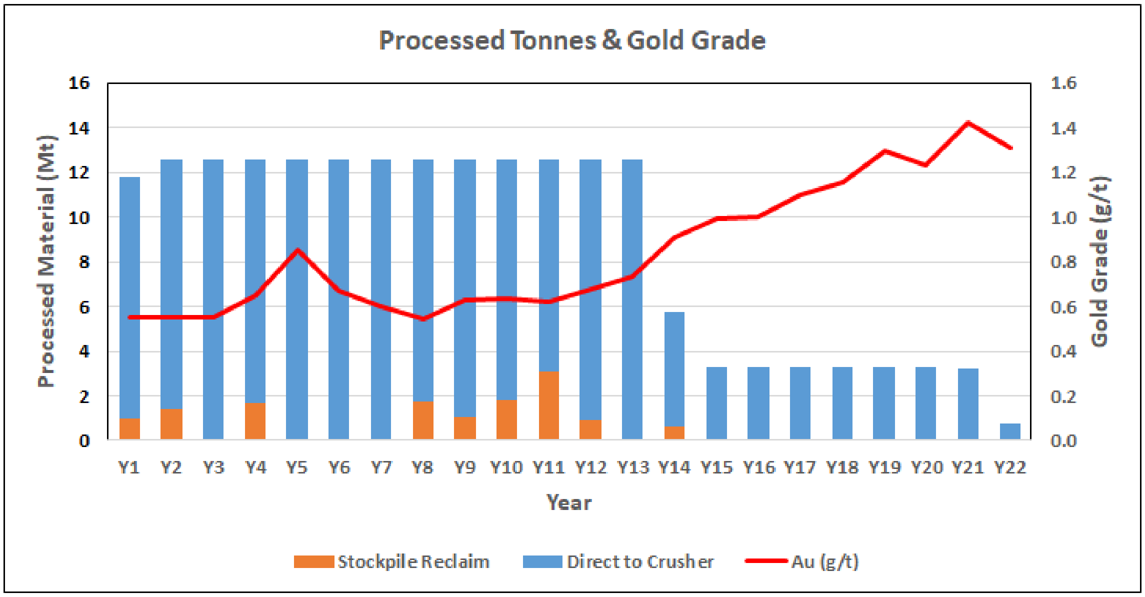


Figure 16-54: Process Tonnage and Gold Grade



The detailed mine schedule was completed on an annual basis and is shown in Table 16-31.

Table 16-31: Mine Schedule

		Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Mining Summary	Waste (Mt)	36.4	62.1	61.1	56.4	58.1	53.9	48.1	37.4	45.6	37.6	35.6
	Mill Feed (Mt)	3.1	10.9	11.2	13.6	11.9	16.1	16.9	12.6	10.9	11.6	10.8
	AuEq (g/t)	0.60	0.61	0.69	0.64	0.77	0.88	0.70	0.70	0.67	0.75	0.81
	Au (g/t)	0.52	0.52	0.57	0.54	0.67	0.74	0.59	0.60	0.58	0.65	0.68
	Ag (g/t)	0.87	0.93	1.19	1.12	1.16	1.09	0.97	1.04	0.93	0.98	1.02
	Cu (%)	0.054	0.060	0.078	0.070	0.070	0.091	0.071	0.066	0.063	0.068	0.084
	Total (Mt)	39.5	73.0	72.3	70.0	70.0	70.0	65.0	50.0	56.5	49.2	46.4
Processed Material	Mill Feed (Mt)	0.0	11.8	12.6	12.6	12.6	12.6	12.6	12.6	12.6	12.6	12.6
	AuEq (g/t)	0.00	0.64	0.66	0.65	0.76	0.99	0.78	0.70	0.64	0.73	0.76
	Au (g/t)	0.00	0.55	0.55	0.55	0.65	0.85	0.67	0.60	0.55	0.63	0.64
	Ag (g/t)	0.00	0.94	1.15	1.15	1.17	1.13	1.05	1.04	0.91	0.97	0.99
	Cu (%)	0.000	0.061	0.074	0.070	0.071	0.094	0.072	0.066	0.064	0.068	0.082
Stockpile Balance	Low Grade (Mt)	2.4	2.2	0.7	1.7	1.1	4.6	8.9	8.8	7.2	6.1	4.3
	AuEq (g/t)	0.44	0.44	0.44	0.45	0.41	0.46	0.46	0.46	0.46	0.46	0.46
	Au (g/t)	0.37	0.37	0.37	0.37	0.36	0.36	0.36	0.36	0.36	0.36	0.36
	Ag (g/t)	0.78	0.78	0.78	0.80	0.52	0.84	0.79	0.79	0.79	0.79	0.79
	Cu (%)	0.046	0.046	0.046	0.055	0.028	0.068	0.069	0.069	0.068	0.068	0.068
	High Grade (Mt)	0.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	AuEq (g/t)	1.11	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Au (g/t)	1.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Ag (g/t)	1.17	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Cu (%)	0.081	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
Stockpile Additions	(Mt)	3.1	0.0	0.0	1.0	1.1	3.5	4.3	0.0	0.1	0.0	0.0
Stockpile Reclaim	(Mt)	0.0	0.9	1.4	0.0	1.7	0.0	0.0	0.1	1.7	1.0	1.8
Material Movement	(Mt)	39.5	73.9	73.7	70.0	71.7	70.0	65.0	50.1	58.3	50.2	48.2

		Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20	Y21	Y22	Total
Mining Summary	**Waste (Mt)**	**29.2**	**20.2**	**8.5**	**0.7**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**591**
	Mill Feed (Mt)	**9.5**	**12.0**	**12.6**	**5.1**	**3.3**	**3.3**	**3.3**	**3.3**	**3.3**	**3.3**	**3.2**	**0.8**	**192**
	AuEq (g/t)	0.84	0.81	0.85	1.12	1.14	1.16	1.24	1.29	1.44	1.35	1.55	1.42	**0.83**
	Au (g/t)	0.70	0.69	0.74	0.98	0.99	1.00	1.10	1.15	1.30	1.23	1.43	1.31	**0.71**
	Ag (g/t)	1.11	1.11	1.06	1.15	0.79	0.75	0.23	0.39	0.15	0.10	0.07	0.01	**0.97**
	Cu (%)	0.088	0.079	0.075	0.091	0.098	0.098	0.089	0.091	0.089	0.077	0.081	0.069	**0.076**
	Total (Mt)	**38.7**	**32.2**	**21.1**	**5.8**	**3.3**	**3.3**	**3.3**	**3.3**	**3.3**	**3.3**	**3.2**	**0.8**	**784**
Processed Material	**Mill Feed (Mt)**	**12.6**	**12.6**	**12.6**	**5.8**	**3.3**	**3.3**	**3.3**	**3.3**	**3.3**	**3.3**	**3.2**	**0.8**	**192**
	AuEq (g/t)	0.74	0.80	0.85	1.04	1.14	1.16	1.24	1.29	1.44	1.35	1.55	1.42	**0.83**
	Au (g/t)	0.62	0.68	0.74	0.91	0.99	1.00	1.10	1.15	1.30	1.23	1.43	1.31	**0.71**
	Ag (g/t)	1.03	1.09	1.06	1.11	0.79	0.75	0.23	0.39	0.15	0.10	0.07	0.01	**0.97**
	Cu (%)	0.083	0.079	0.075	0.088	0.098	0.098	0.089	0.091	0.089	0.077	0.081	0.069	**0.076**
Stockpile Balance	**Low Grade (Mt)**	**1.3**	**0.6**	**0.6**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	
	AuEq (g/t)	0.46	0.44	0.44	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	
	Au (g/t)	0.36	0.35	0.35	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	
	Ag (g/t)	0.79	0.74	0.74	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	
	Cu (%)	0.068	0.062	0.062	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	
	High Grade (Mt)	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	
	AuEq (g/t)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	
	Au (g/t)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	
	Ag (g/t)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	
	Cu (%)	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	
Stockpile Additions	**(Mt)**	**0.0**	**0.3**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**13.4**
Stockpile Reclaim	**(Mt)**	**3.1**	**0.9**	**0.0**	**0.6**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**13.4**
Material Movement	**(Mt)**	**41.8**	**33.1**	**21.1**	**6.5**	**3.3**	**3.3**	**3.3**	**3.3**	**3.3**	**3.3**	**3.2**	**0.8**	**797**

Mineralized material in the production schedule was split into low-grade and high-grade groups using a diluted gold cut-over grade of 0.7 g/t. The stockpiled mill feed, together with pit phasing, will be utilized to ensure mill feed is available during the transition to phases which require considerable pre-stripping of waste.

Table 16-32 displays a summary of the resource classifications for the mill feed.

Table 16-32: Resource Summary of Scheduled Material

Mining Type	Area	Indicated					Inferred				
		Mill Feed (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	AuEq (g/t)	Mill Feed (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	AuEq (g/t)
Open Pit	Z87	34.7	0.73	0.090	1.40	0.86	1.9	0.55	0.063	1.39	0.64
	J	79.3	0.52	0.060	0.90	0.61	15.5	0.45	0.062	0.82	0.54
	SW	0.0	0.00	0.000	0.00	0.00	18.8	0.64	0.065	0.76	0.74
	Subtotal	**114.0**	**0.59**	**0.069**	**1.05**	**0.69**	**36.2**	**0.56**	**0.064**	**0.82**	**0.65**
Underground	Z87	30.1	1.17	0.111	1.08	1.34	12.2	1.21	0.093	0.33	1.36
Total		**144.1**	**0.71**	**0.078**	**1.06**	**0.82**	**48.4**	**0.72**	**0.071**	**0.70**	**0.83**

Year -1 has mining initiated in phase 1 of both the SW and Z87 pits. In this time period, a total of 36.4 Mt of waste material will be moved as the project ramps up. The mill feed stockpile pad is established during this period so that 3.1 Mt of mill feed can be sent to the mill stockpile grading 0.52 g/t Au, 0.054% Cu and 0.87 g/t Ag in anticipation of plant commissioning and operation. 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. The SW dyke is completed during this period to establish the diversion ditch along the west of the project. Overburden material is directed to dedicated storage areas for each pit. Rock waste is sent from active mining areas to the TSF buttress lifts from year -1 to year 9.

Year 1 production assumes the plant will require three months to achieve full production levels. The first month the plant will be capable of 65% of capacity, the second month 75%, and the third month 85%. Subsequent months will be at 100% of the 35 ktpd nameplate capacity in the mill. This plant ramp-up schedule requires the Year 1 production to be 11.8 Mt. Mill feed will be from stockpile, SW phase 1, and Z87 phase 1. J phase 1 was started at the end of year 1.

Year 2 production is at the full 12.6 Mt of mill feed. SW phase 1 mining is completed with its final level being 5240 masl. Z87 phase 1 and J phase 1 continue to be mined down to the levels of 5220 masl, and 5340 masl, respectively. The J pit waste storage areas for rock and overburden are both established resulting in the final waste footprint for the project. All future waste destinations involve either adding lifts or the backfilling of the Z87 pit.

Year 3 mining continues in Z87 phase 1 and J phase 1 down to the levels of 5160 masl and 5290 masl, respectively. Mining is started in SW phase 2 and advanced down to 5370 masl level. This is the final year that Z87 pit waste is sent to the 87-WD.

Year 4 mining continues in Z87 phase 1, J phase 1 and SW phase 2 down to the levels of 5100 masl, 5270 masl, and 5320 masl, respectively. Mining is started in J phase 2 and advanced down to 5360 masl level. This is the final year that overburden is sent to the SW overburden dump. All Z87 pit waste is sent to the TSF buttress lifts for years 4 to 6.

Year 5 mining continues in Z87 phase 1, J phase 1, J phase 2 and SW phase 2 down to the levels of 5040 masl, 5260 masl, 5310 masl and 5260 masl, respectively. This period marks the end of the high mining rates with total mining production rates of 70 Mtpa. Mining tonnages will begin to decline over the next few years.

Year 6 is a significant year as the Z87 open pit mining is completed to the pit bottom level of 5030 masl. This milestone allows short hauls to be used to backfill the Z87 pit. The Z87 underground portal is established near the mill crusher and development is started. The mining continues in J phase 1, J phase 2 and SW phase 2 down to the levels of 5210 masl, 5250 masl and 5200 masl, respectively. With the Z87 backfill destination becoming available, this is the final period where the J-WD receives waste. Years 6 to 9 will require waste rock from J pit to be sent to the TSF buttress lifts.

Year 7 is the final year of mining in SW phase 2 and J phase 1 where they reach the levels of 5160 masl and 5170 masl, respectively. It is also the first year of mining in J phase 3 which progressed down to 5360 masl. J phase 2 also continues mining down to the 5190 masl level. No more mining activity is scheduled for the SW mining area after this period with the exception of waste sent to the TSF buttress lifts from J phases. Underground development is progressed with the continued establishment of twin declines for rail-Veyor infrastructure. The majority of the waste from J pit is sent to the Z87 backfill.

Year 8 is the first year with mill feed being supplemented, as 106 kt comes from the Z87 underground mine. The mining continues in J phases 2 and 3 down to the levels of 5130 masl and 5310 masl, respectively. This is the final year with overburden mining in J pit so also marks the end of material being sent to the J overburden dump.

Year 9 is the final year of mining for J phase 2 as it progressed down to 5070 masl. The only other open pit mining continues in J phase 3 down to the level of 5260 masl. Z87 underground mill feed release increases to 2.2 Mt and results in higher grades being processed.

Years 10 to 13 were scheduled with 12.6 Mt of mill feed from J phase 3, Z87 underground and stockpiles. All open pit waste was sent to the Z87 pit backfill during these periods.

Year 14 is the final year of mining in J phase 3 as it reached the level of 4990 masl. The annual mill production was lowered to 5.8 Mt in this period as the stockpiles were also exhausted.

Years 15 to 22 mill feed consists entirely of Z87 underground material. Year 15 to 20 have a scheduled mill production of 3.29 Mt, while years 21 and 22 are reduced to 3.25 Mt and 0.8 Mt, respectively. Z87 underground mining is completed during year 22.

16.5.1 End of Year Plans

End of year positions for the open pits are shown in Figure 16-55 to Figure 16-69 while underground positions are shown in Figure 16-70 to Figure 16-86.

Figure 16-55: End of Pre-Production - Year -1

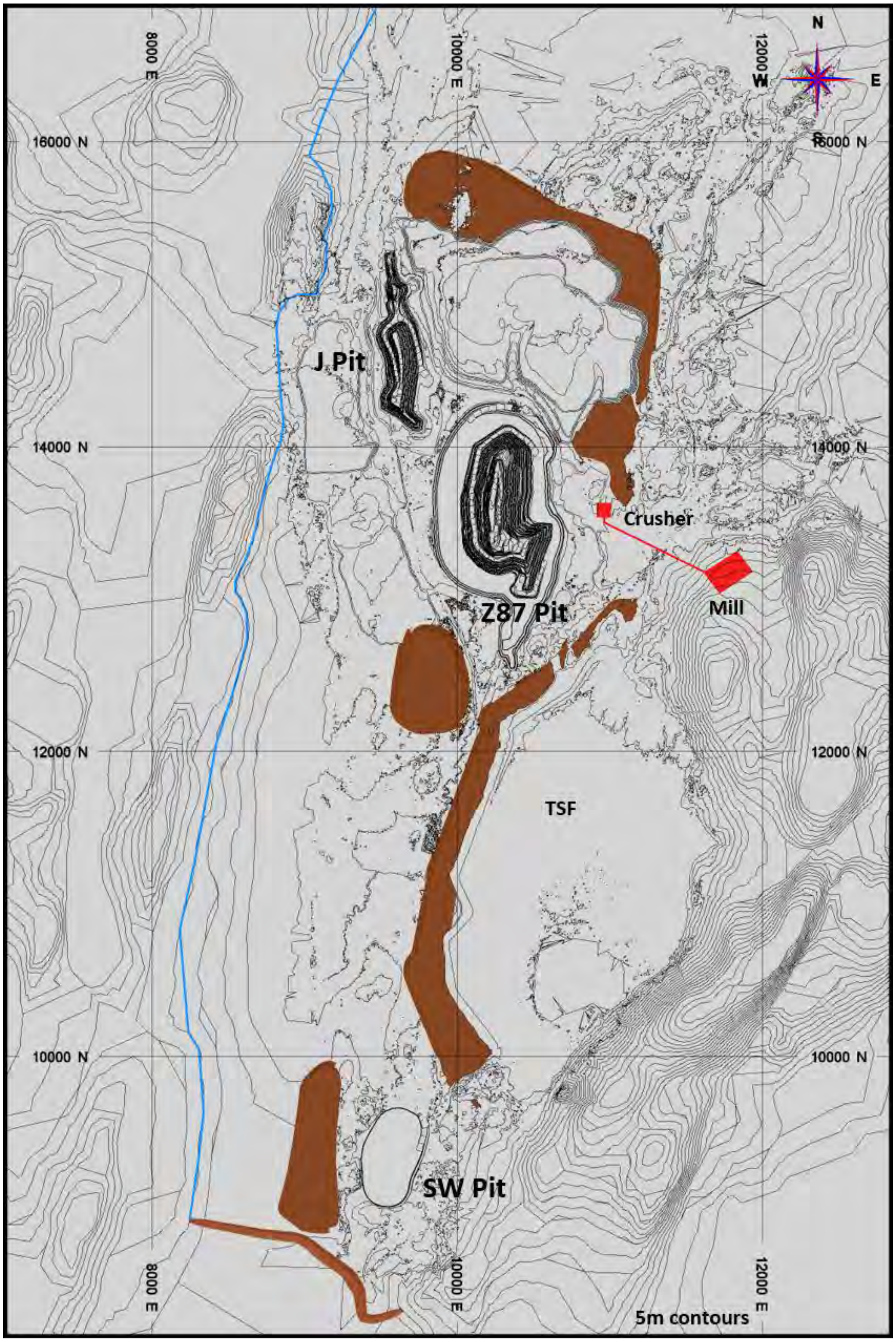


Figure 16-56: End of Year 1

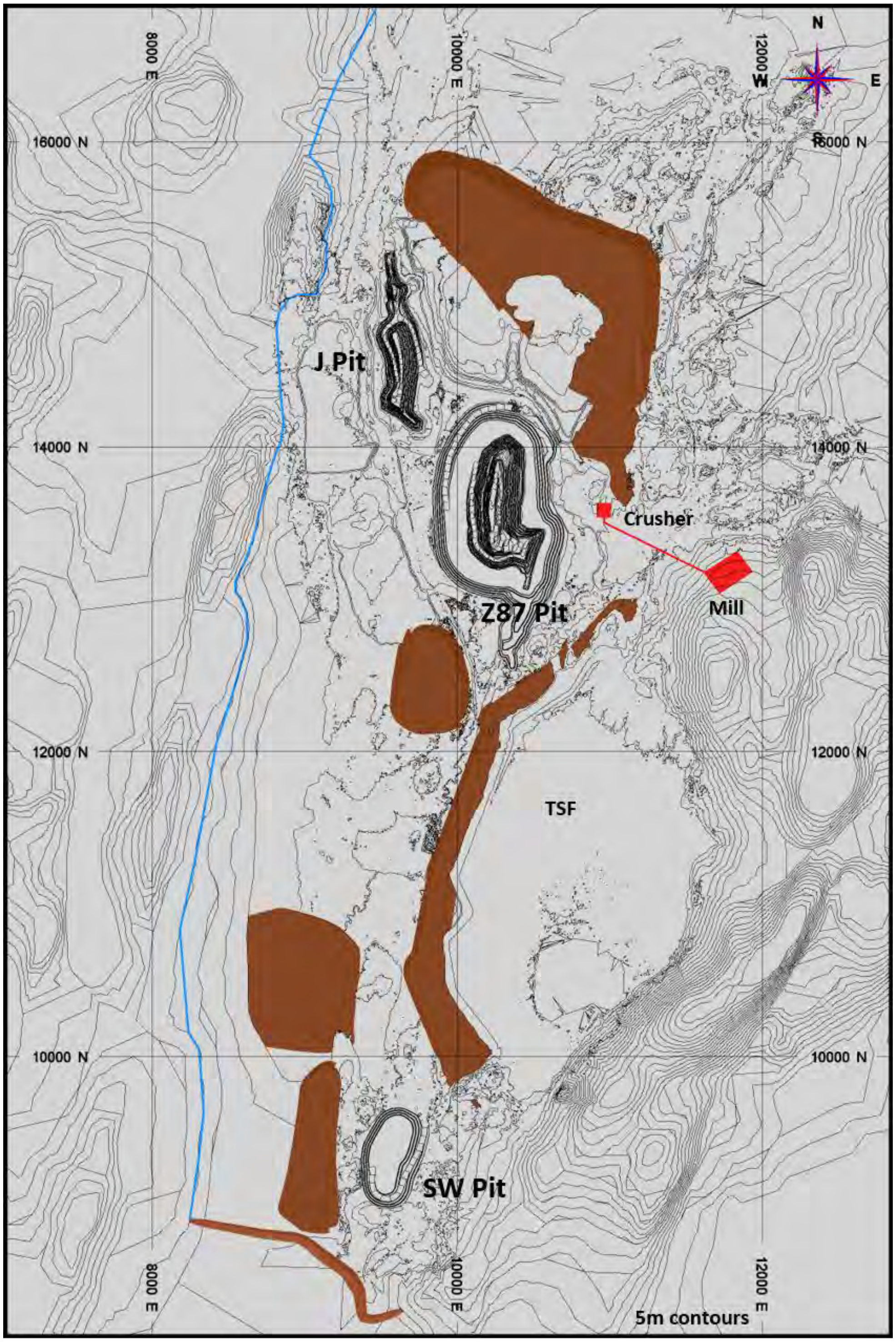


Figure 16-57: End of Year 2

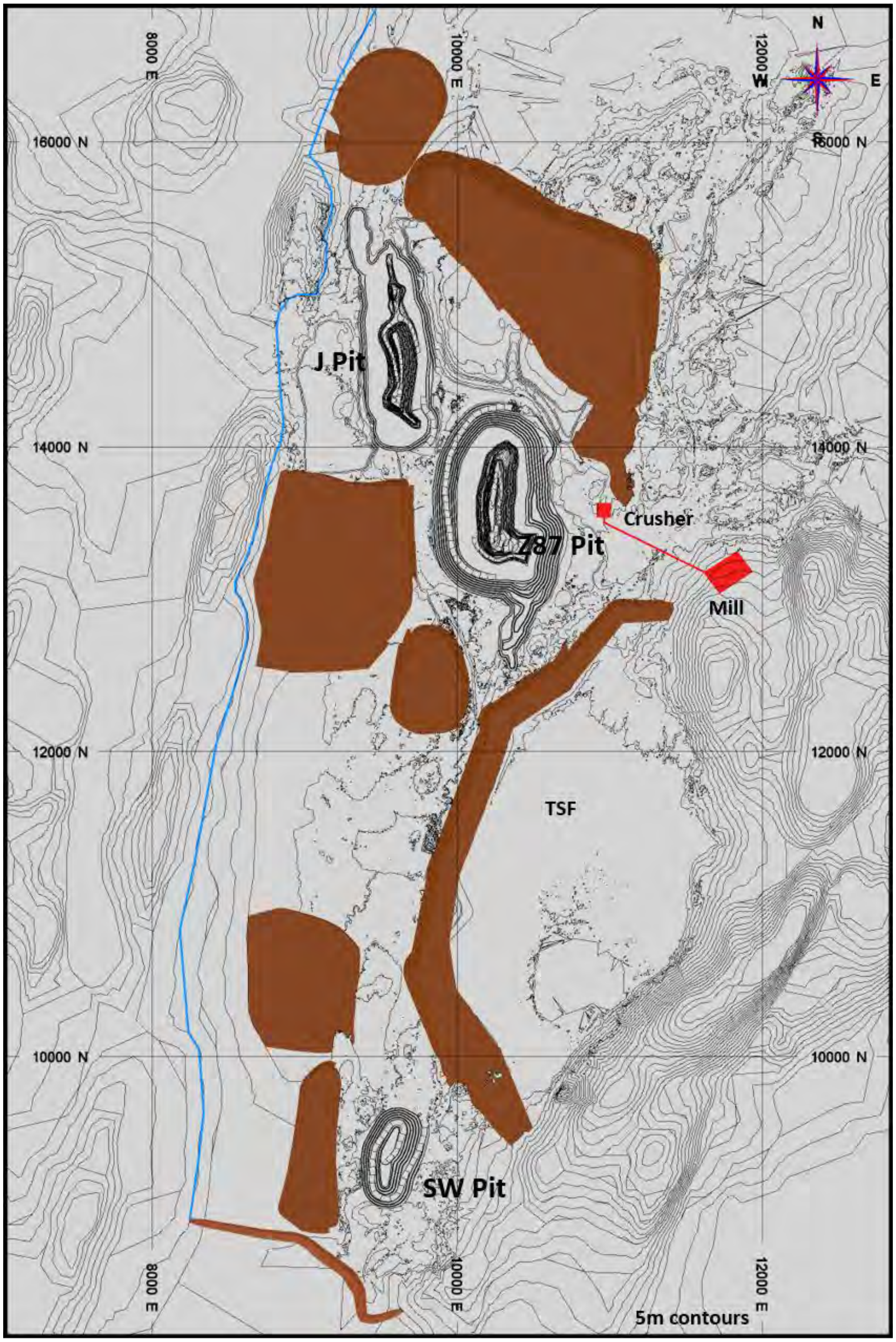


Figure 16-58: End of Year 3

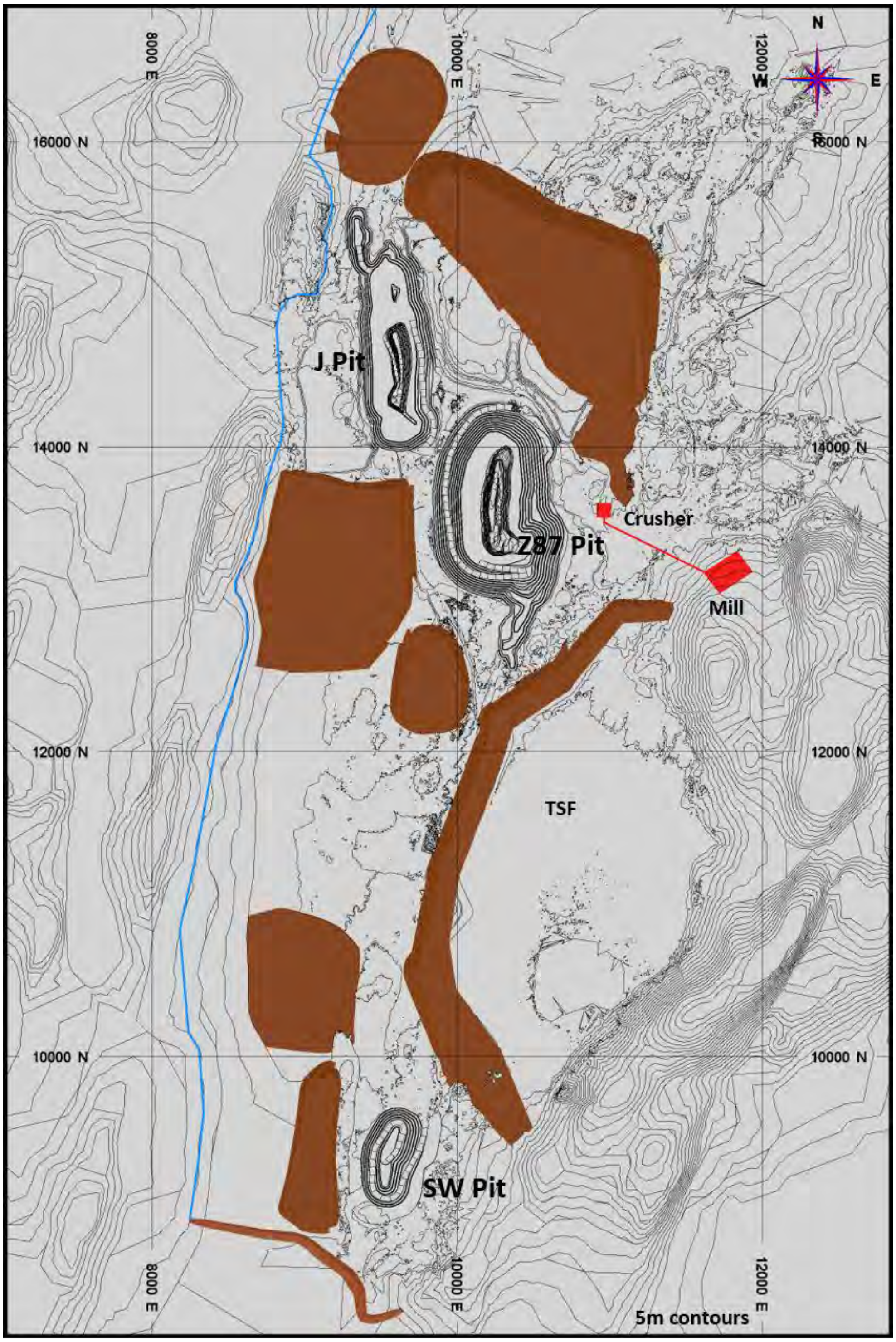


Figure 16-59: End of Year 4

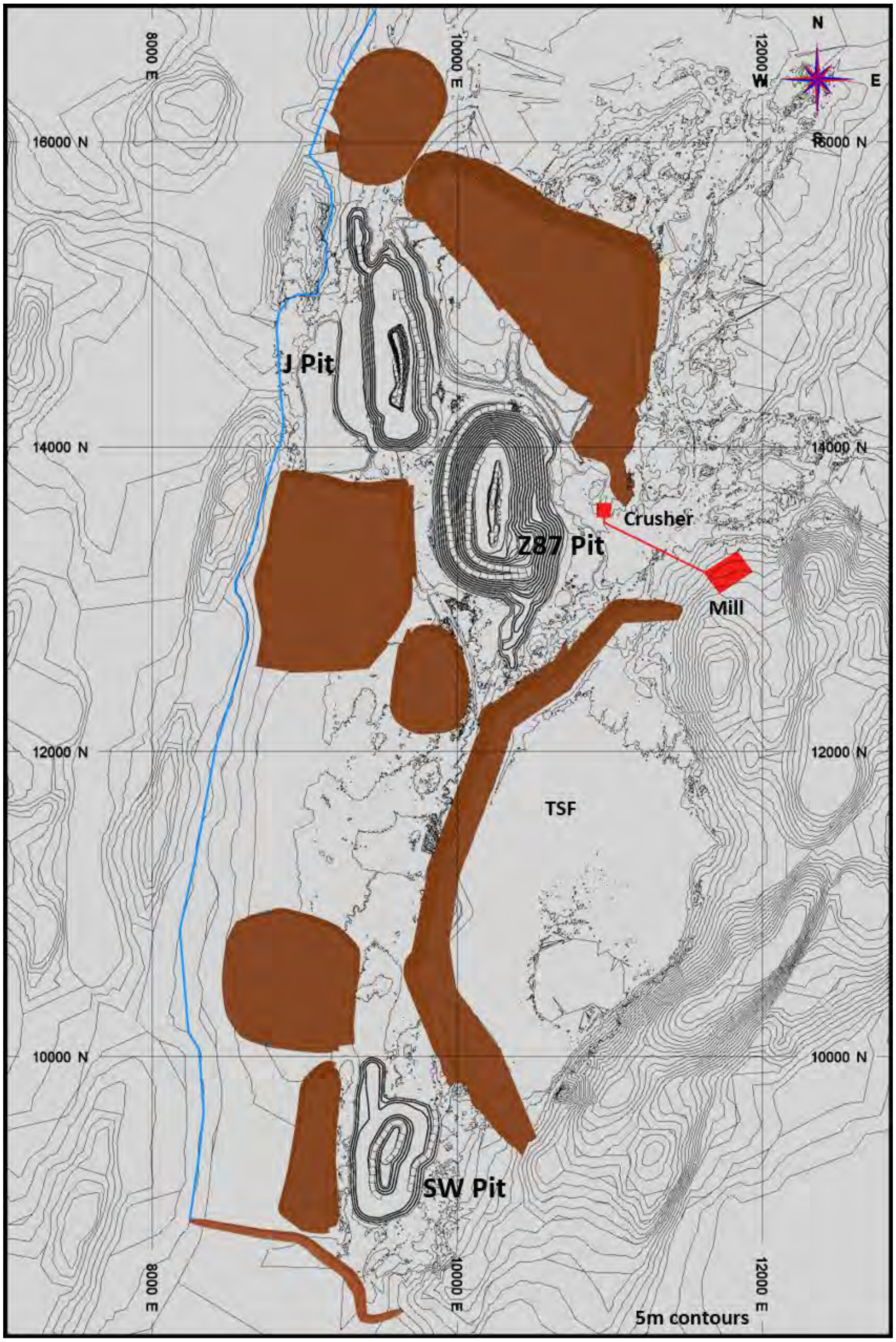


Figure 16-60: End of Year 5

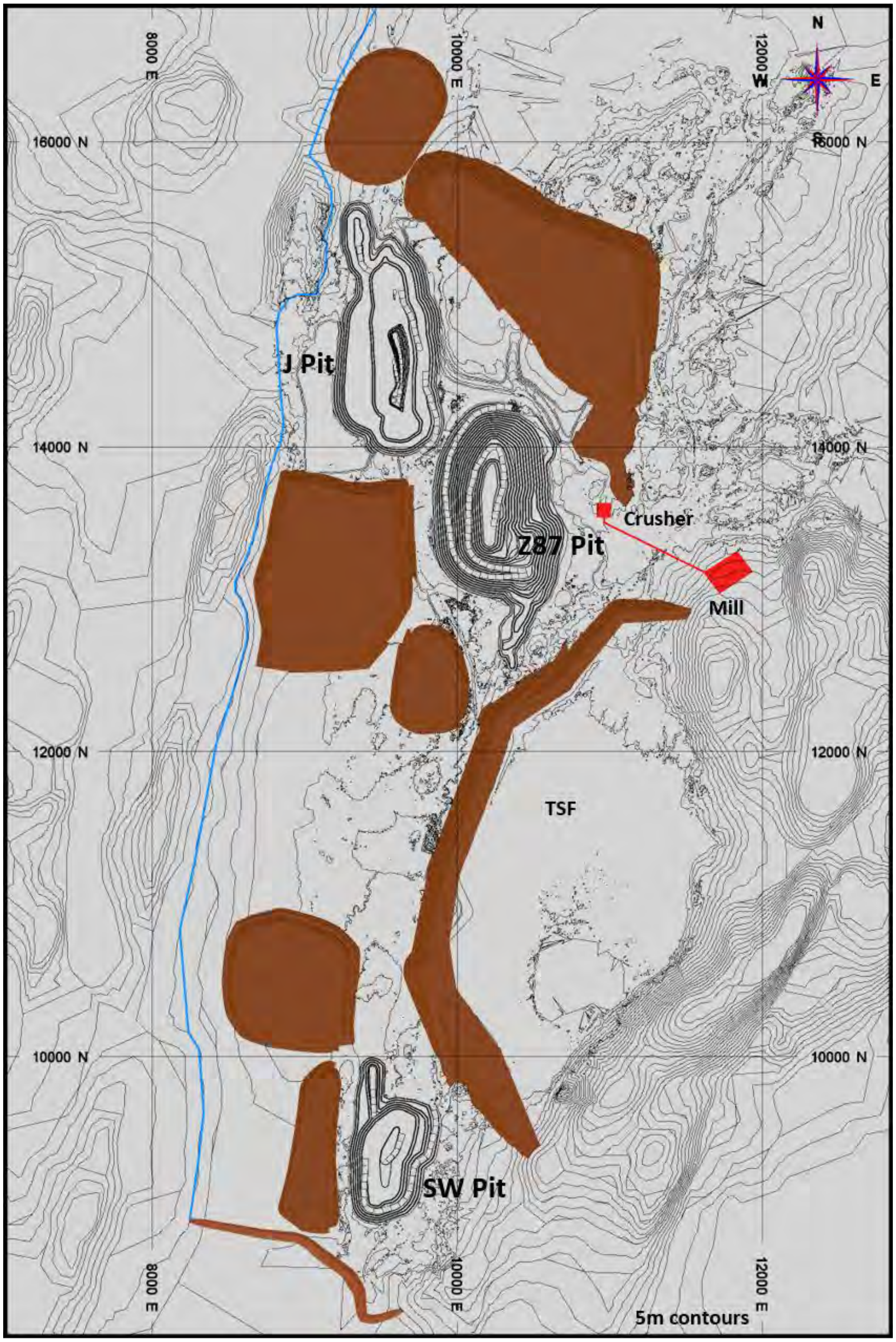


Figure 16-61: End of Year 6

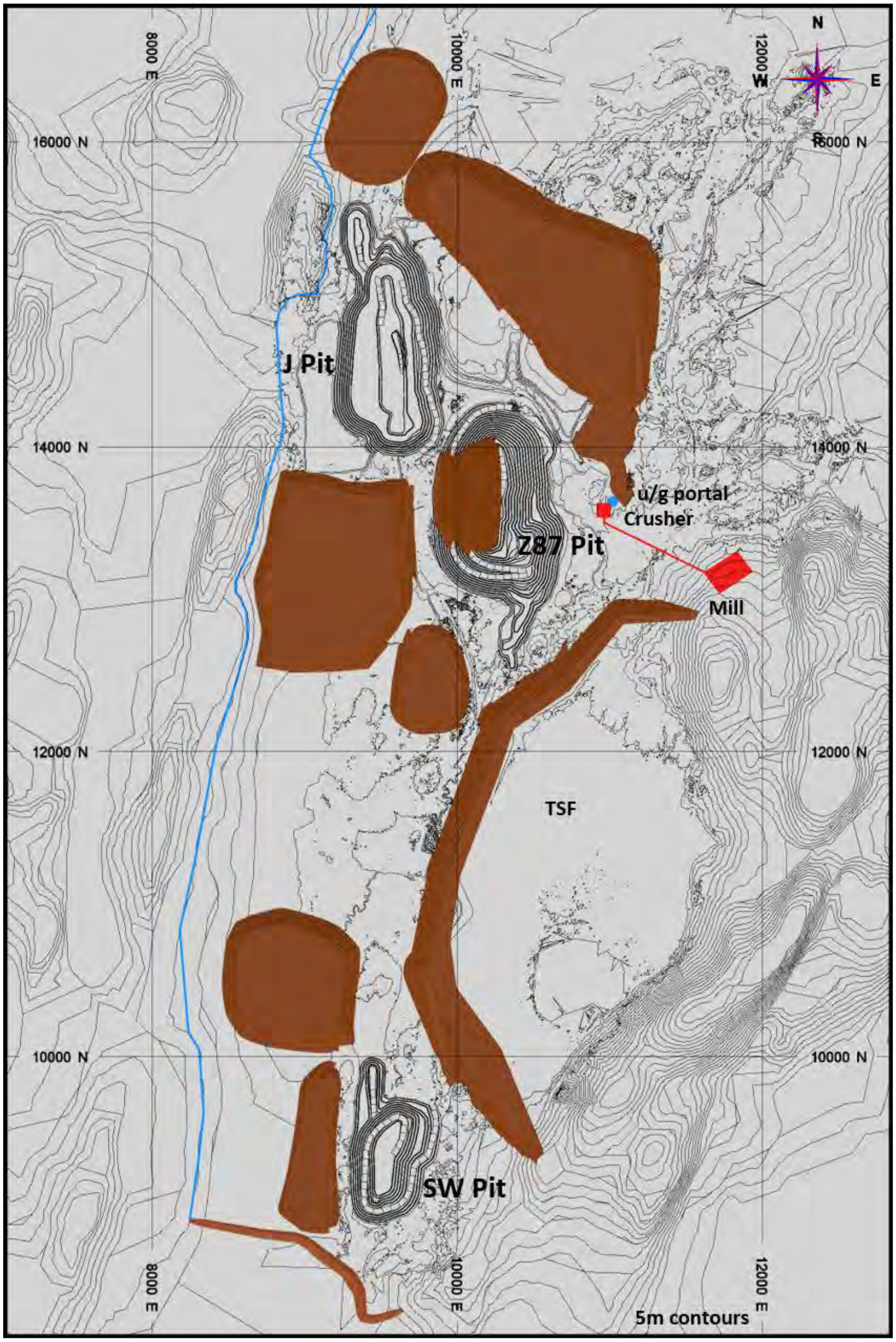


Figure 16-62: End of Year 7

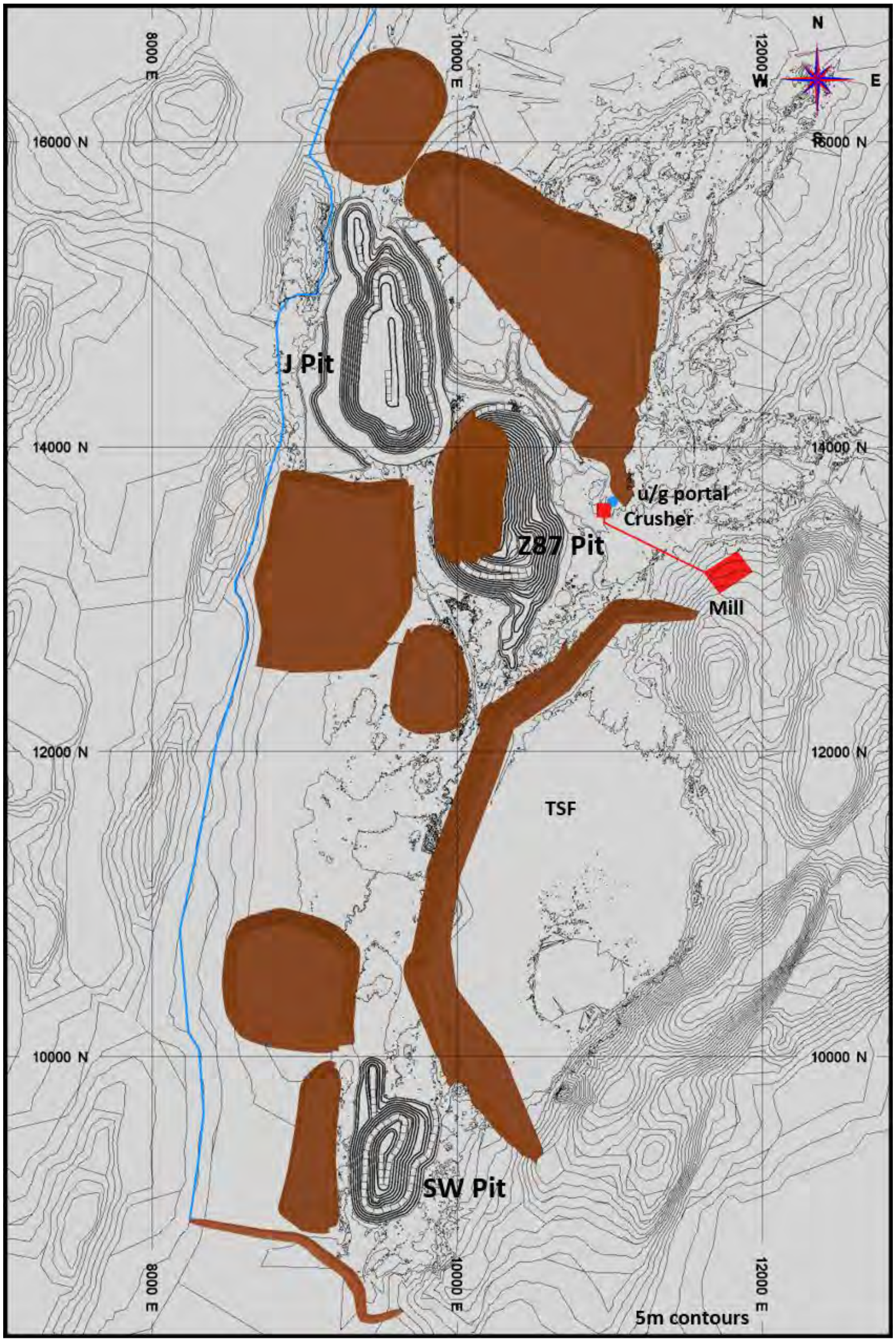


Figure 16-63: End of Year 8

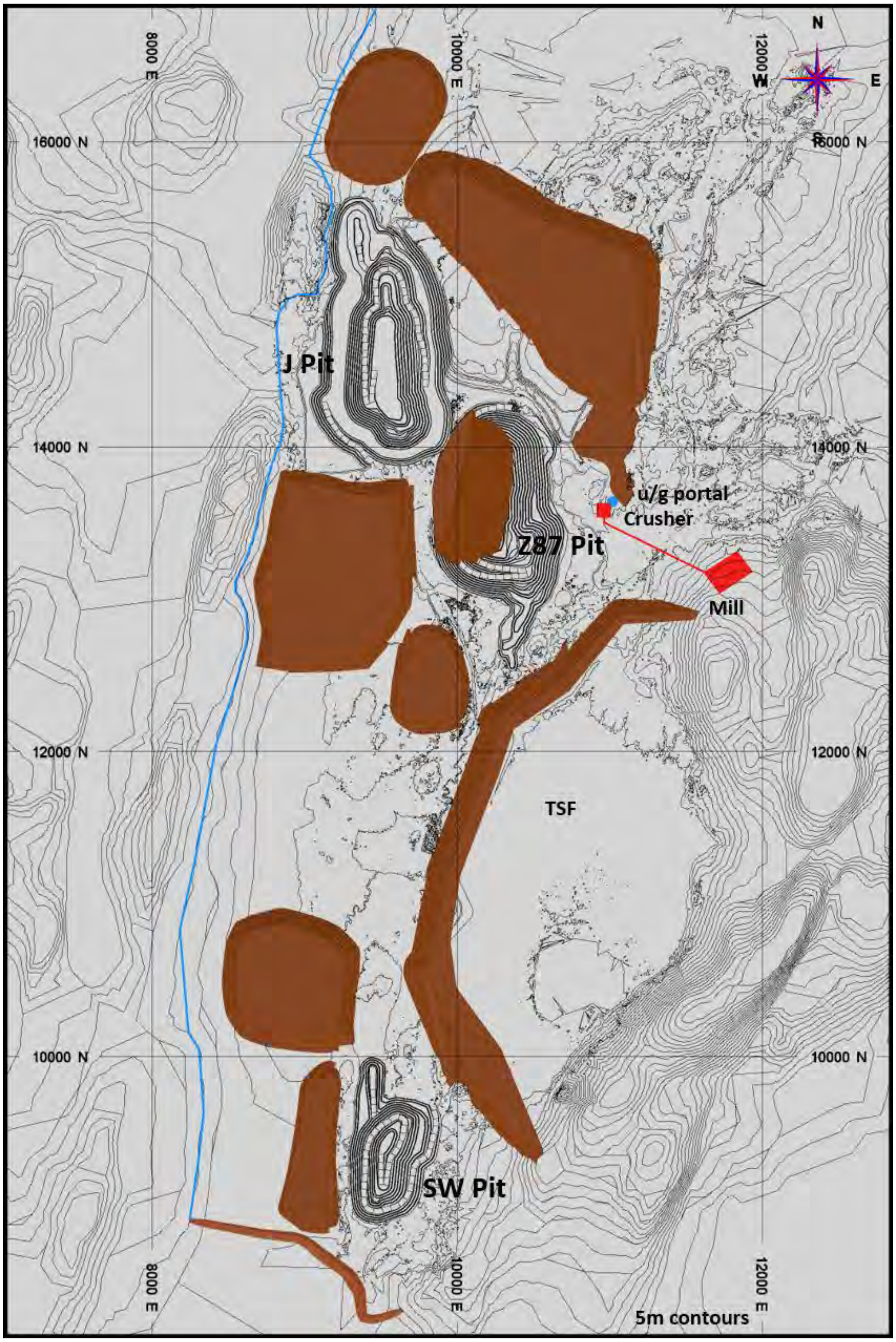


Figure 16-64: End of Year 9

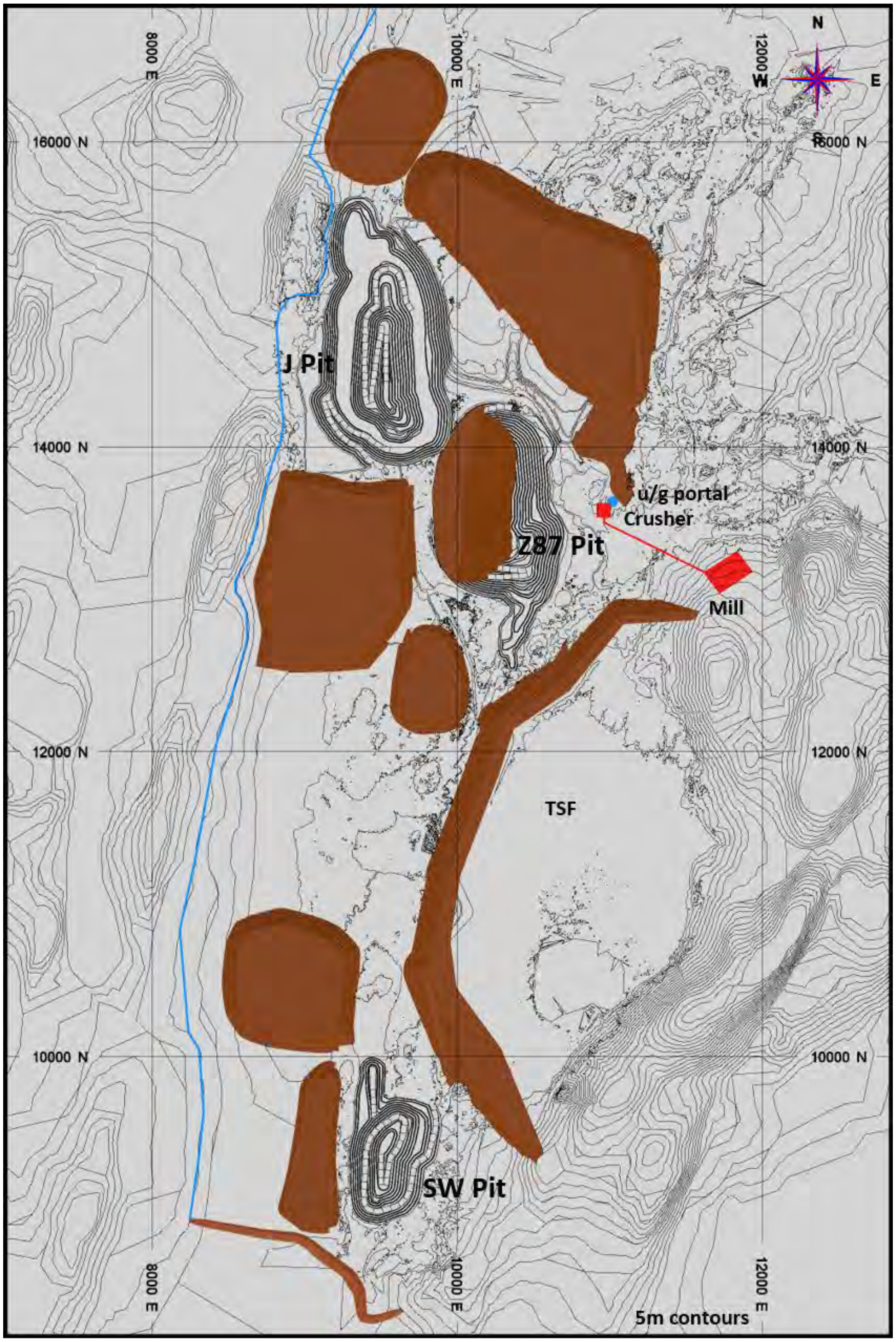


Figure 16-65: End of Year 10

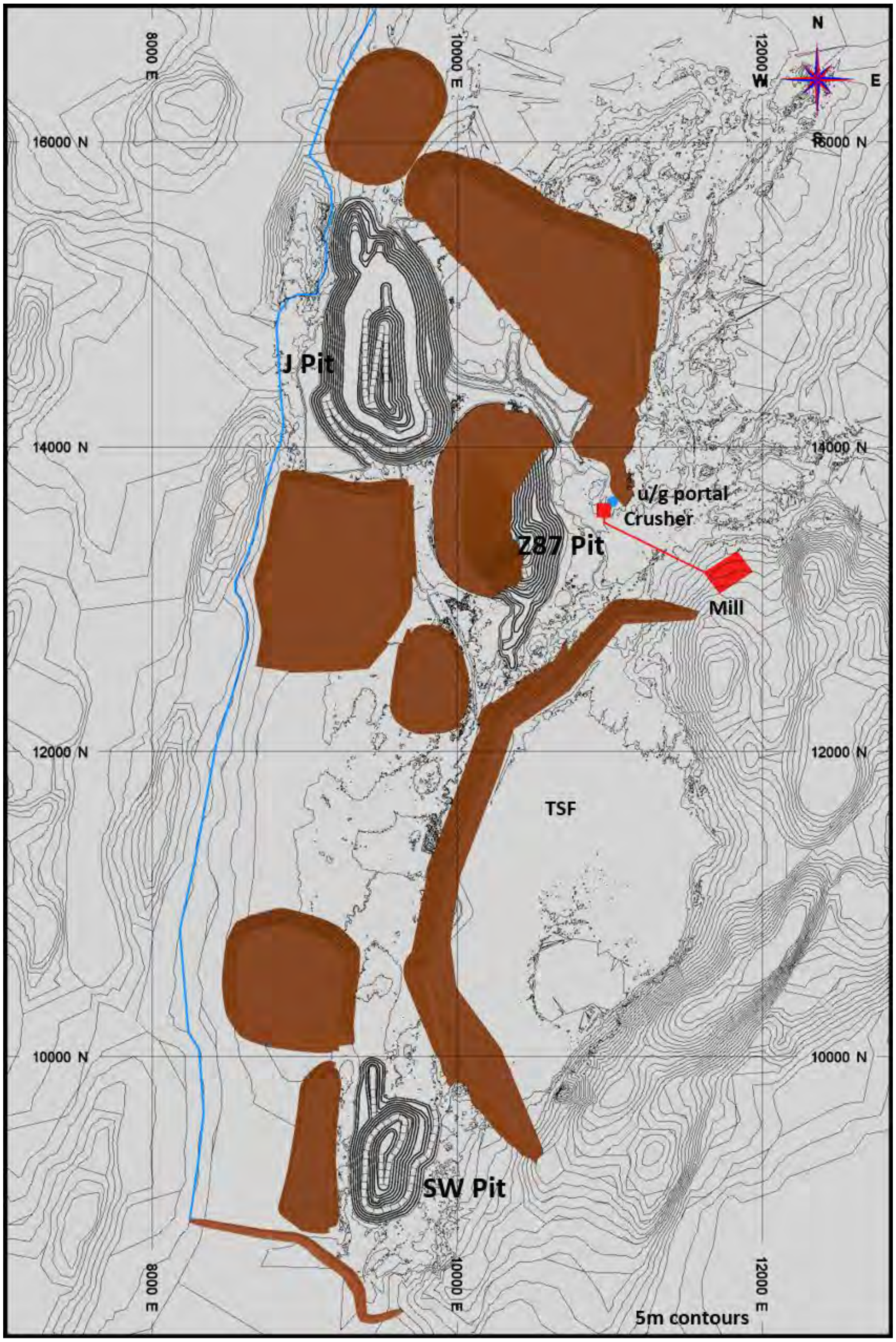


Figure 16-66: End of Year 11

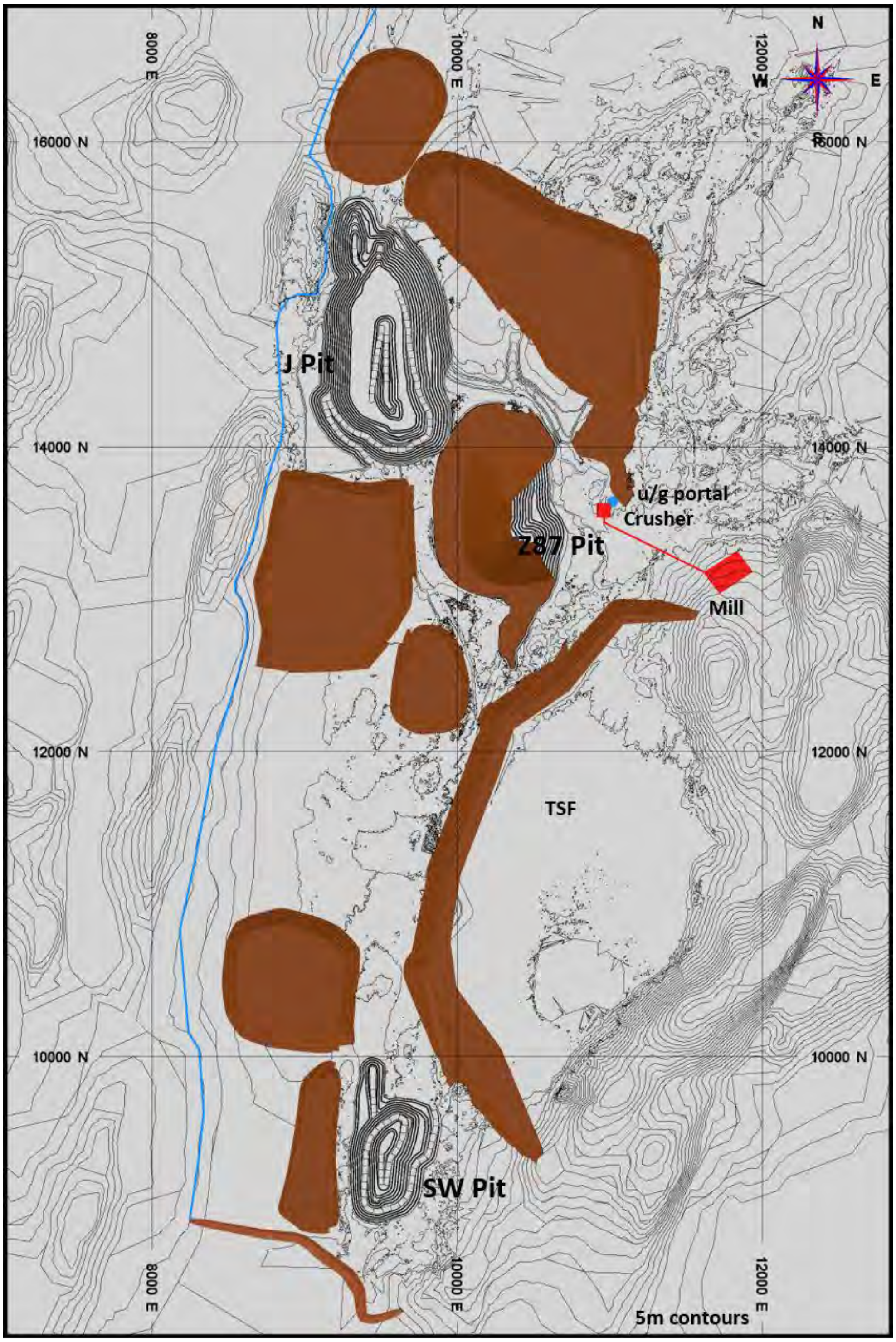


Figure 16-67: End of Year 12

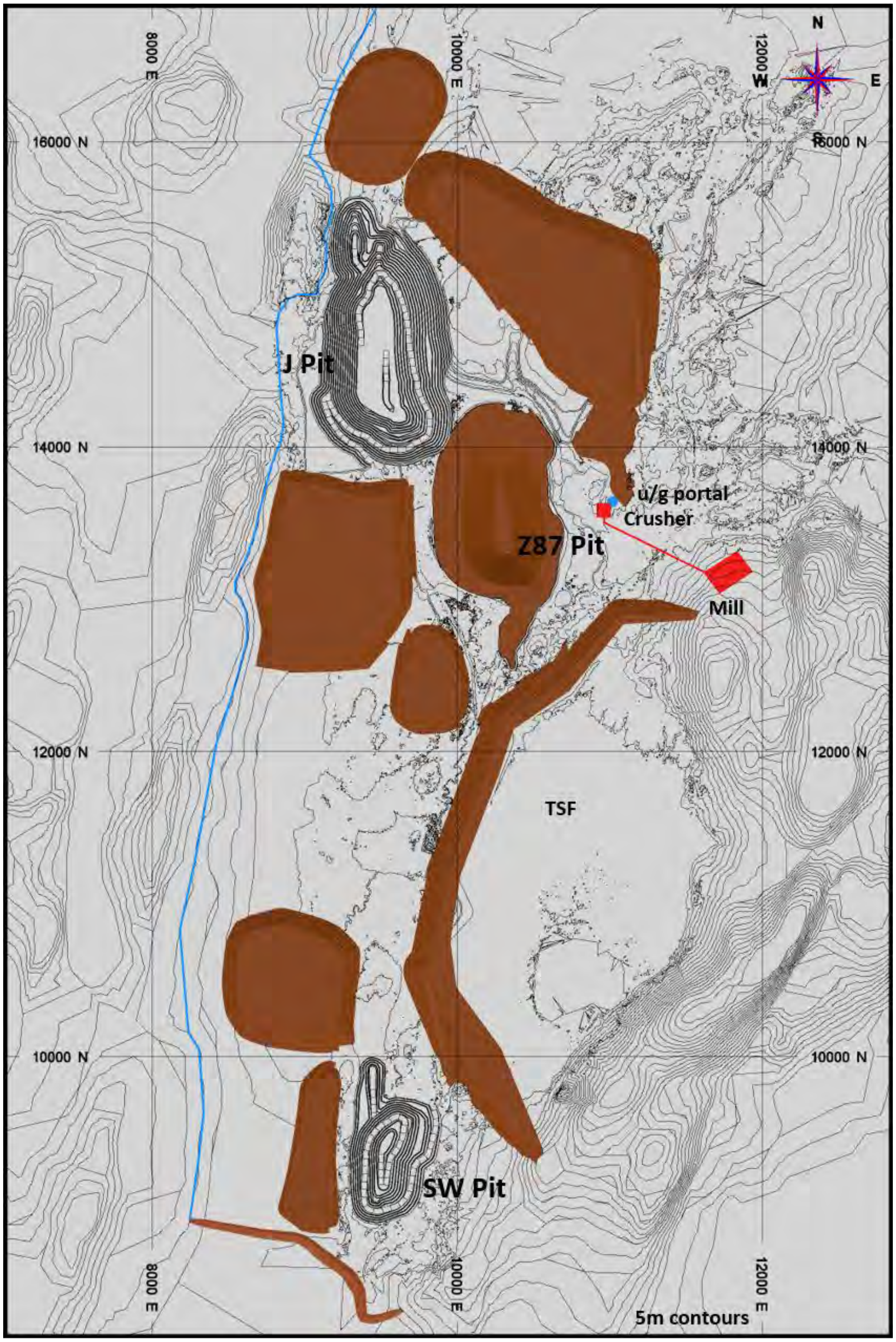


Figure 16-68: End of Year 13

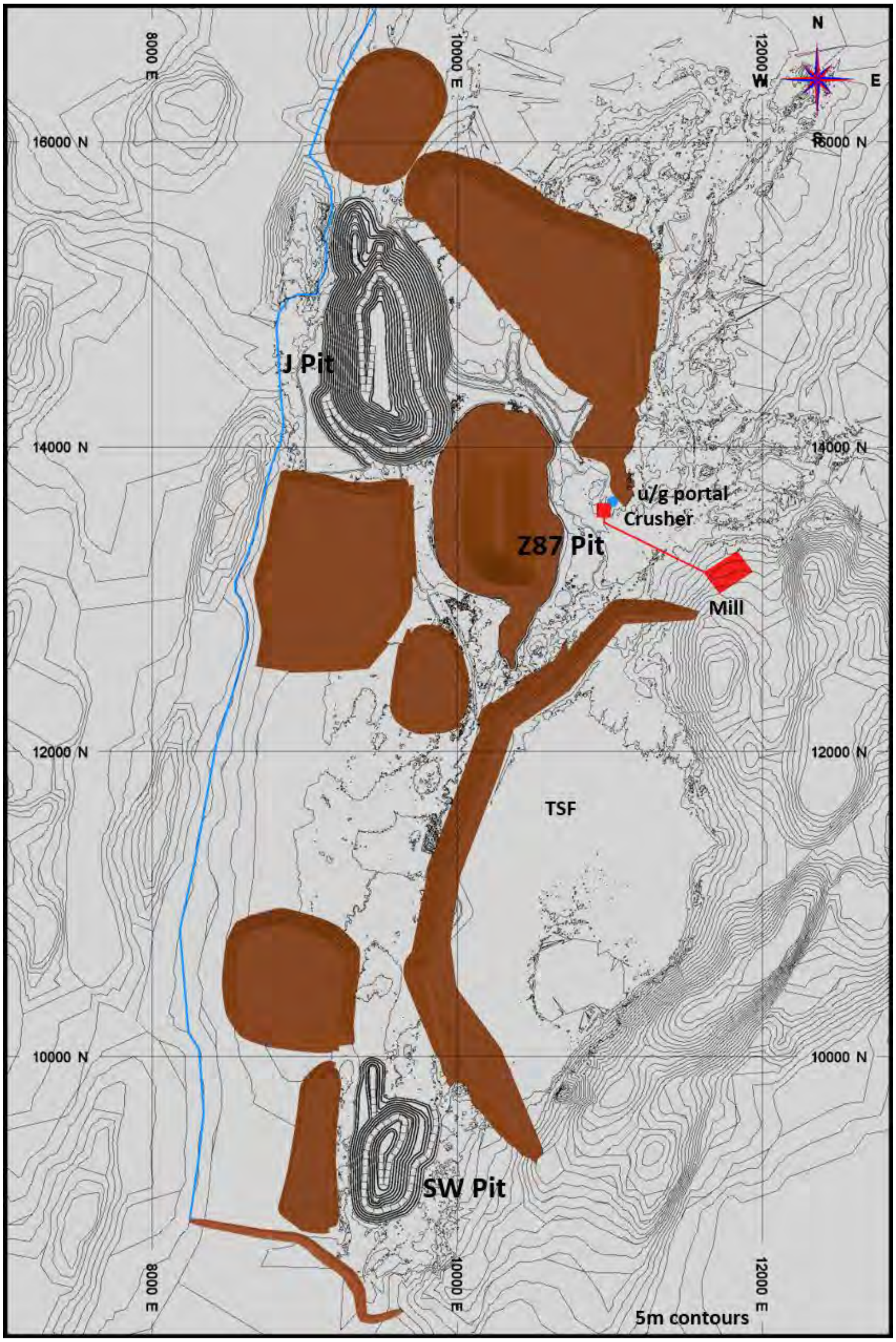


Figure 16-69: End of Year 14

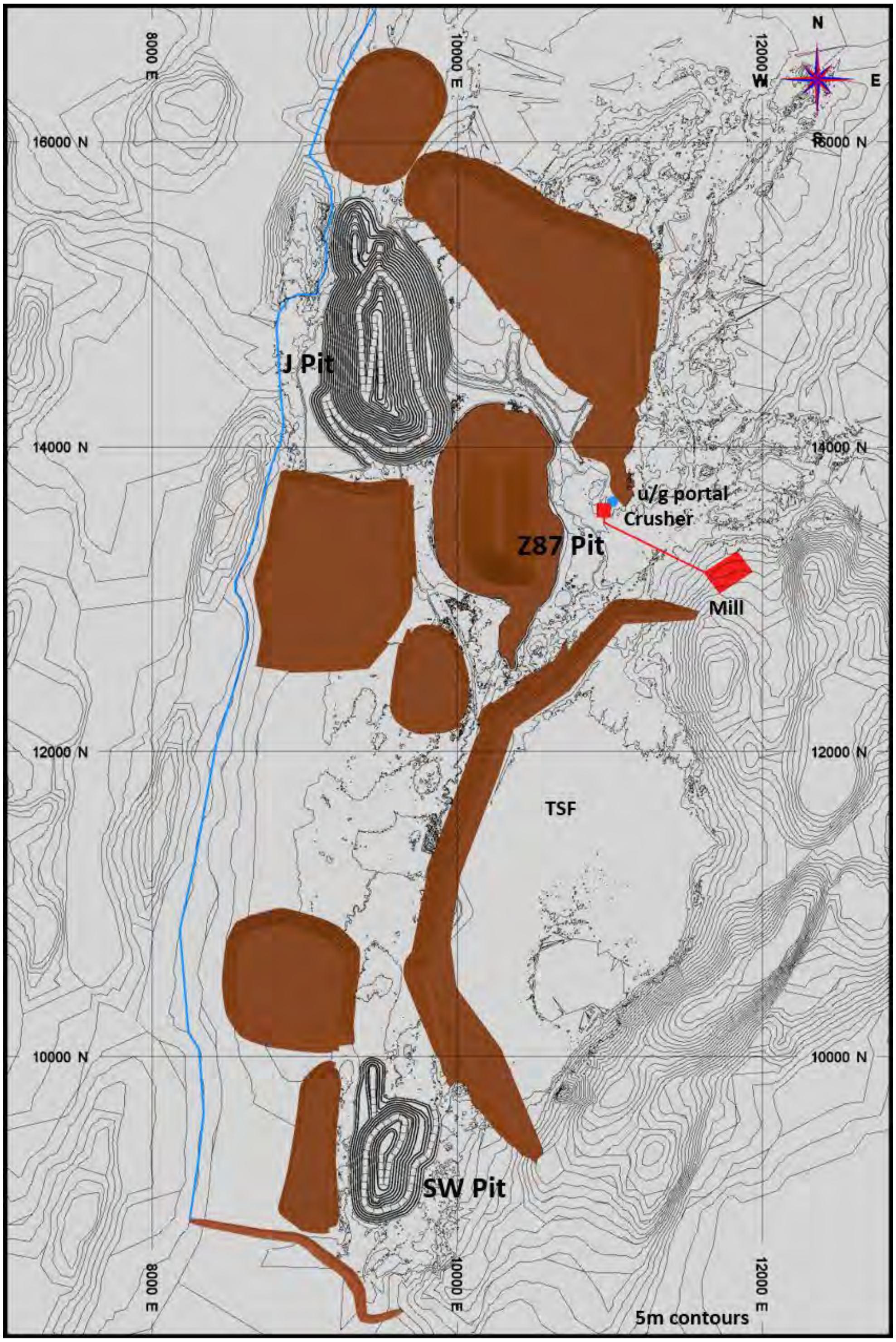


Figure 16-70: End of UG Year 1 (Year 6 of combined schedule)

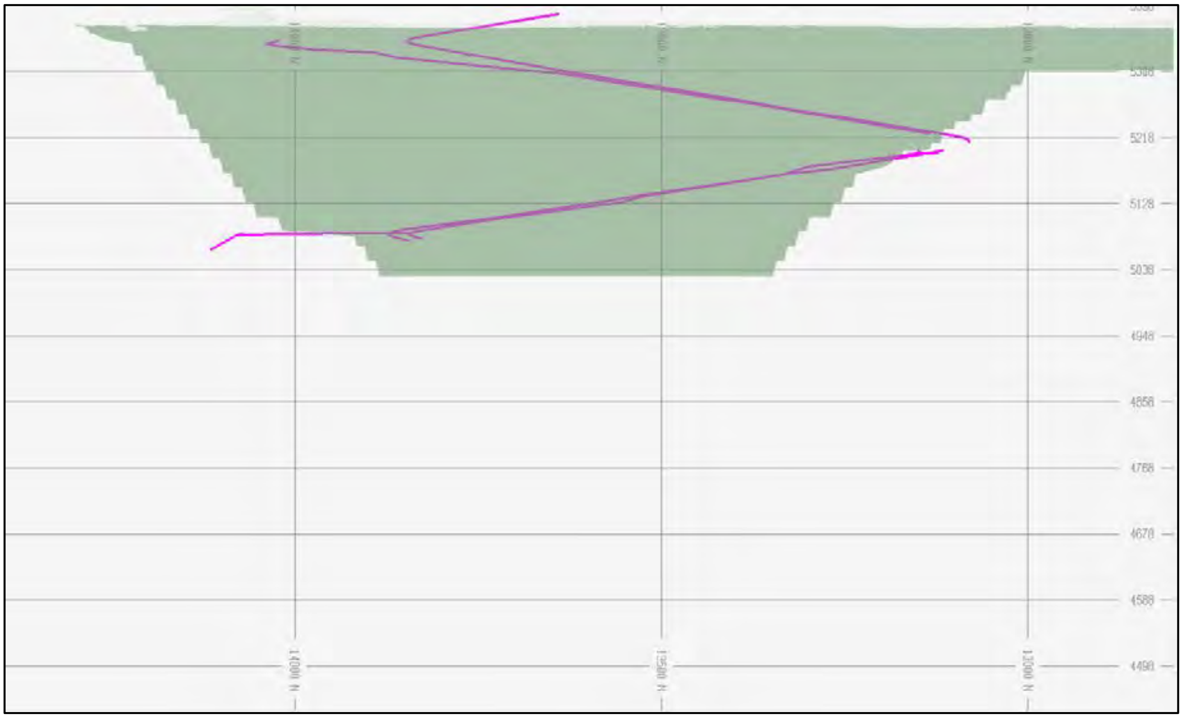


Figure 16-71: End of UG Year 2 (Year 7 of combined schedule)

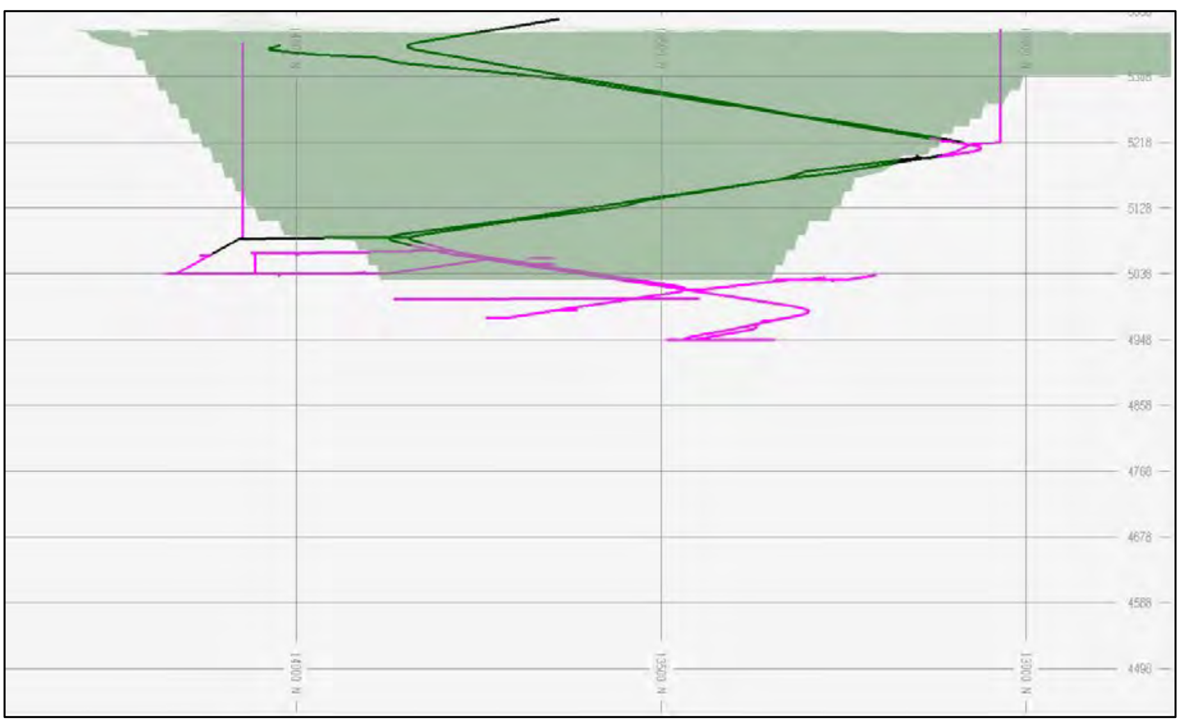


Figure 16-72: End of UG Year 3 (Year 8 of combined schedule)

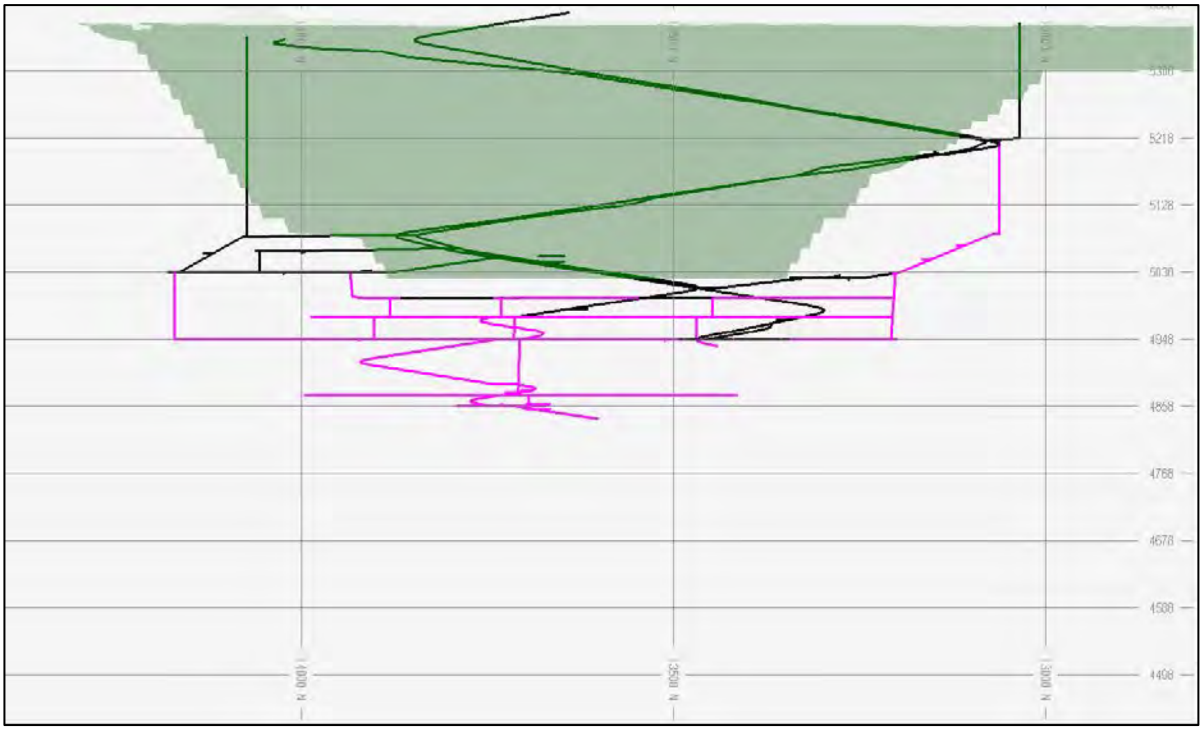


Figure 16-73: End of UG Year 4 (Year 9 of combined schedule)

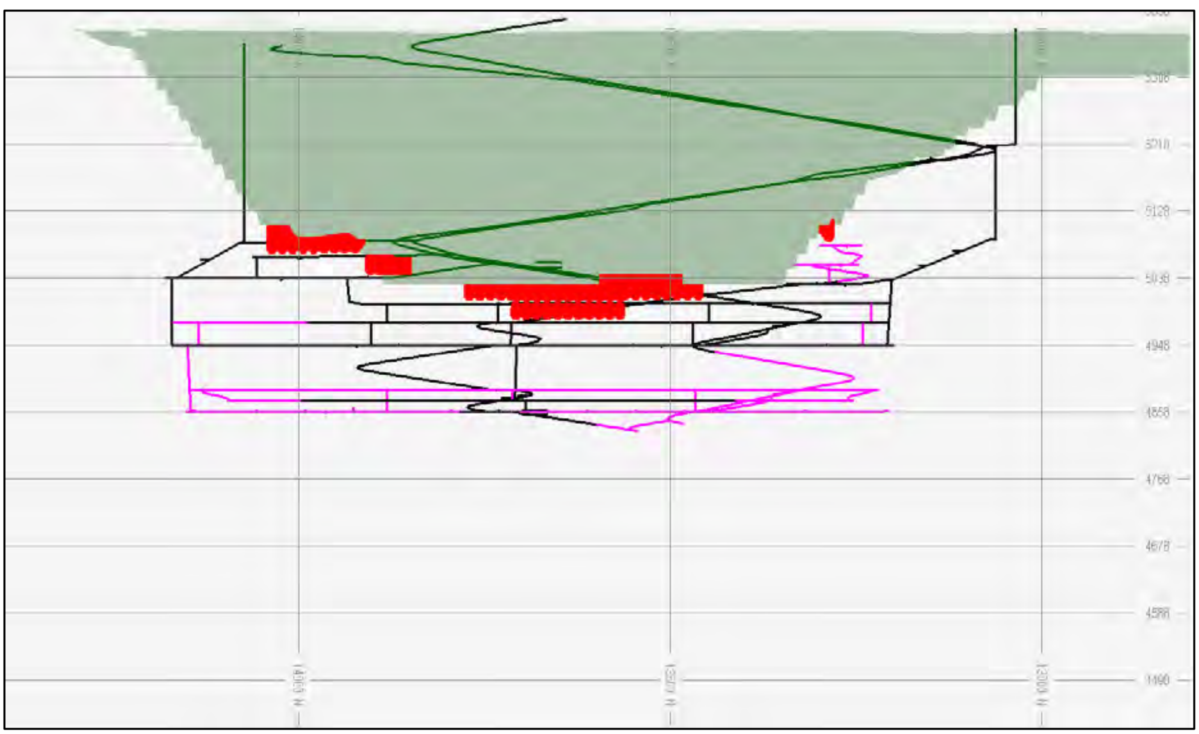


Figure 16-74: End of UG Year 5 (Year 10 of combined schedule)

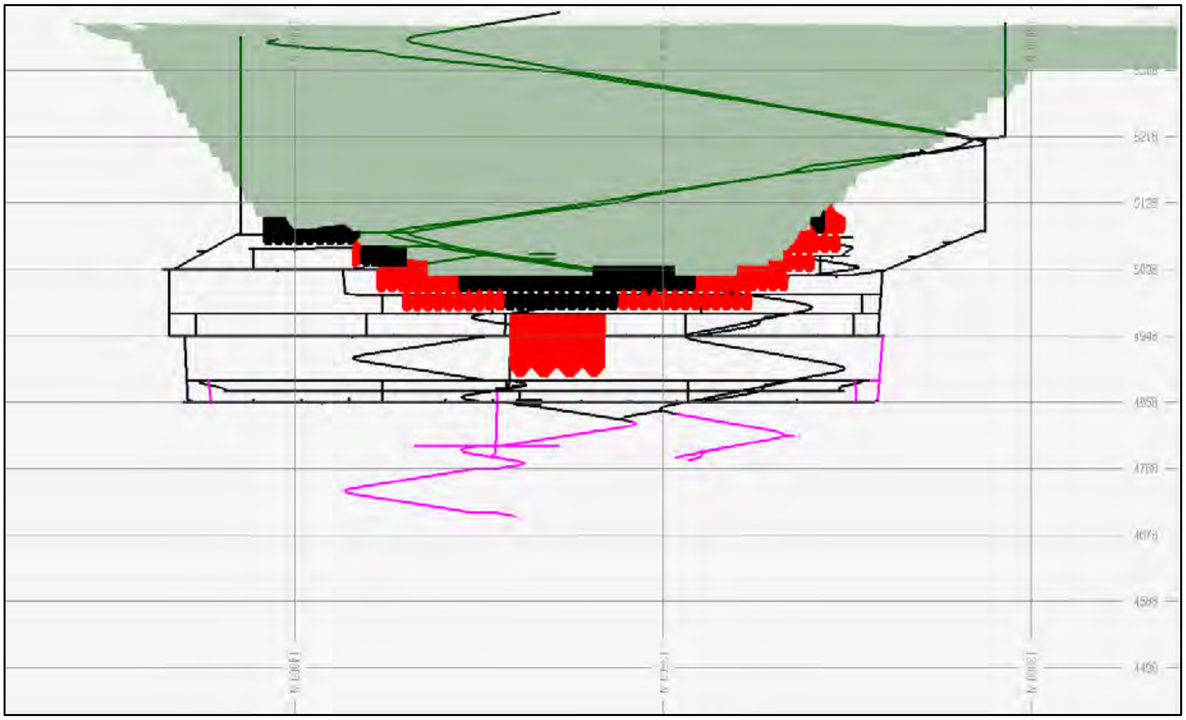


Figure 16-75: End of UG Year 6 (Year 11 of combined schedule)

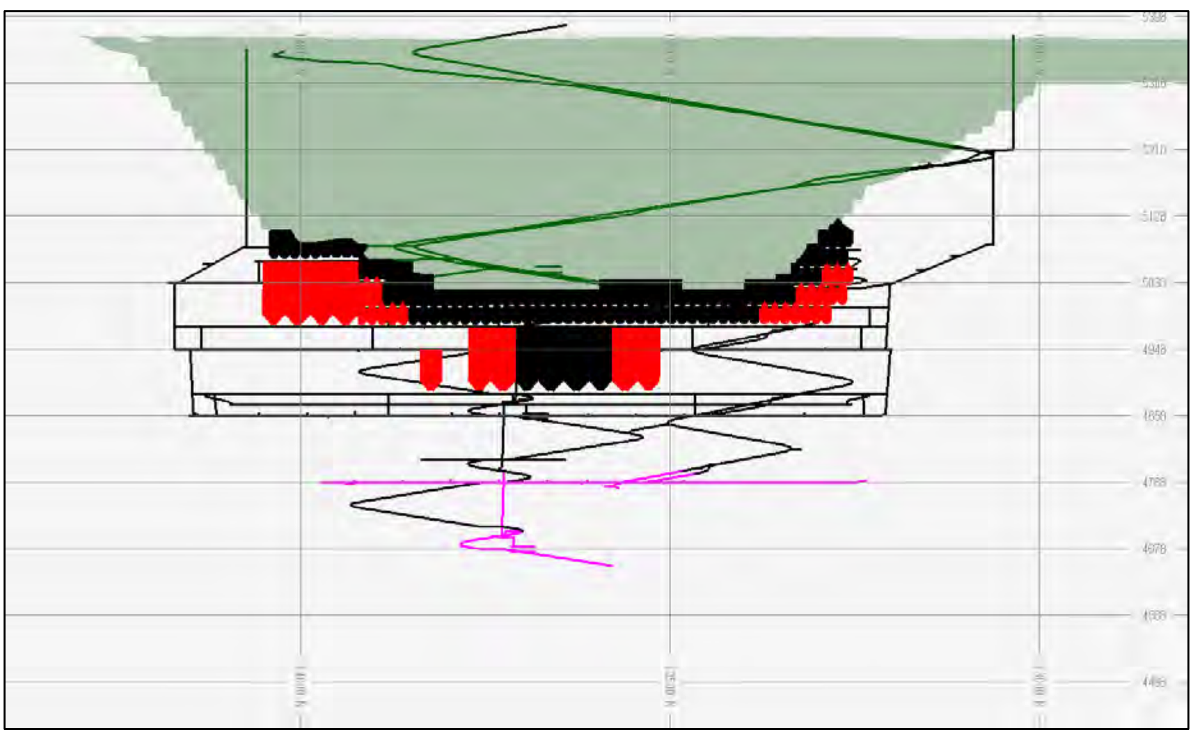


Figure 16-76: End of UG Year 7 (Year 12 of combined schedule)

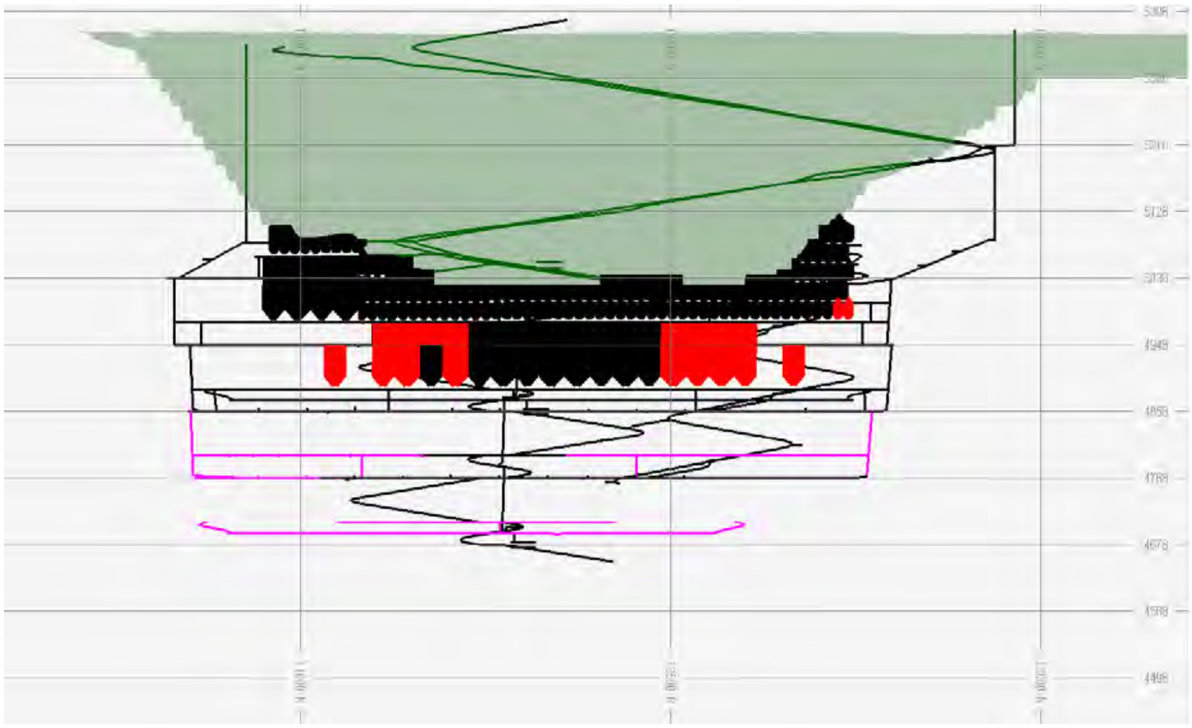


Figure 16-77: End of UG Year 8 (Year 13 of combined schedule)

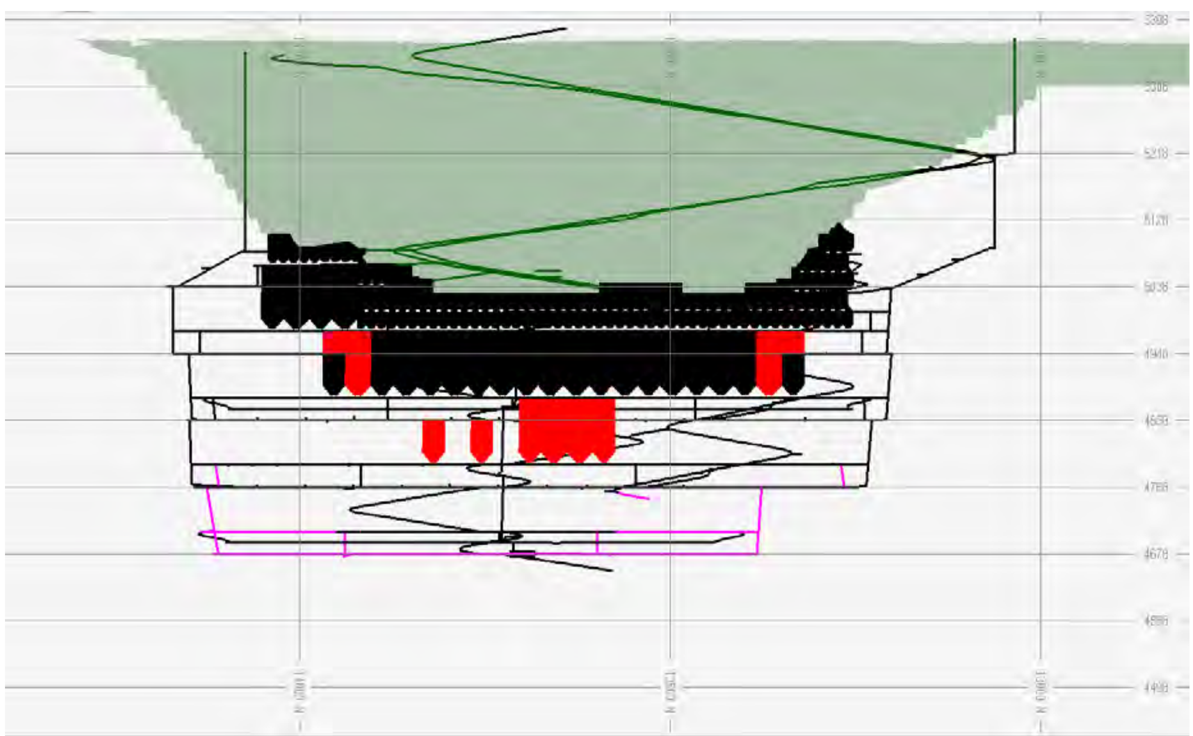


Figure 16-78: End of UG Year 9 (Year 14 of combined schedule)

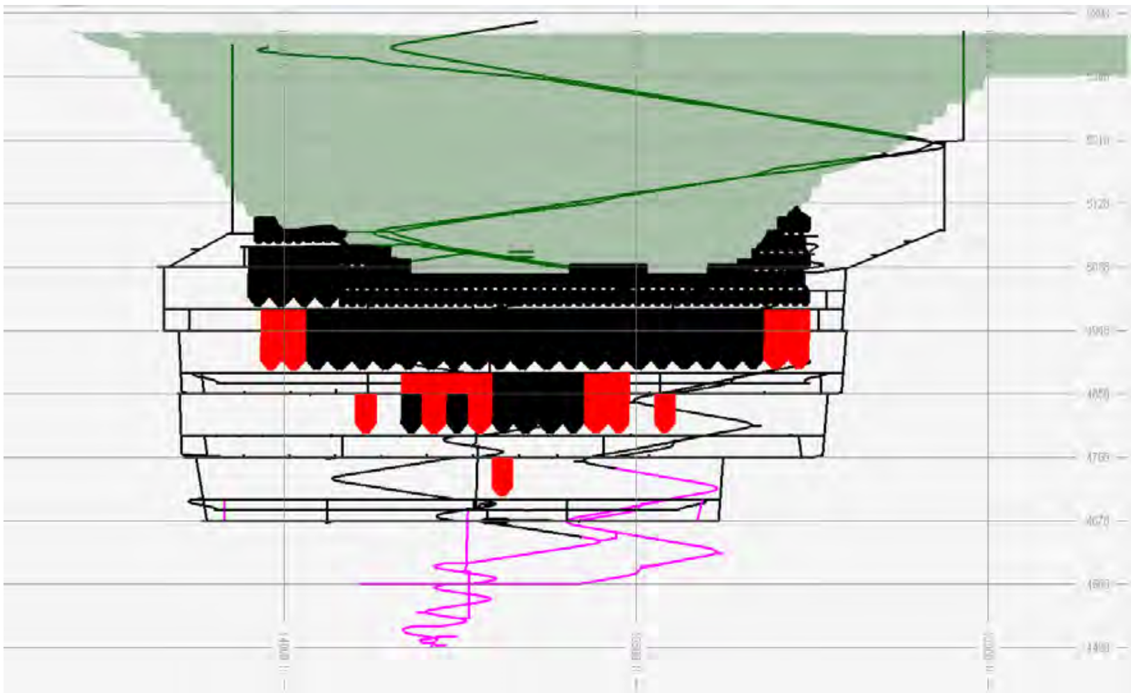


Figure 16-79: End of UG Year 10 (Year 15 of combined schedule)

Figure 16-80: End of UG Year 11 (Year 16 of combined schedule)

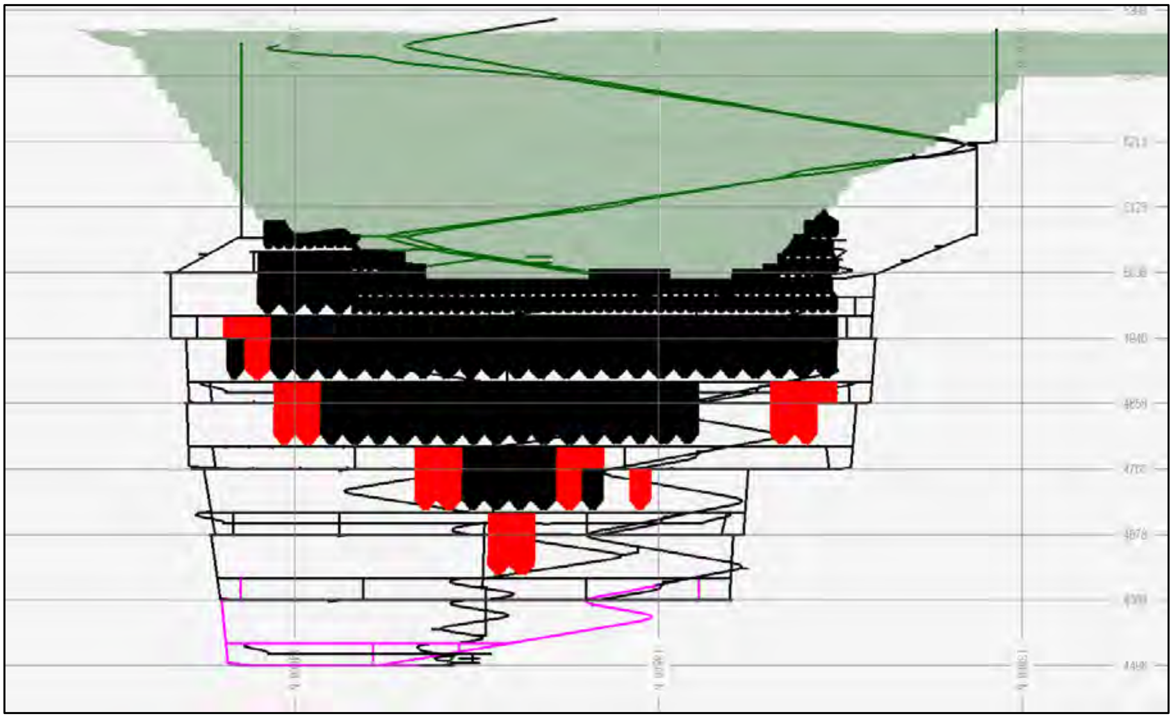


Figure 16-81: End of UG Year 12 (Year 17 of combined schedule)

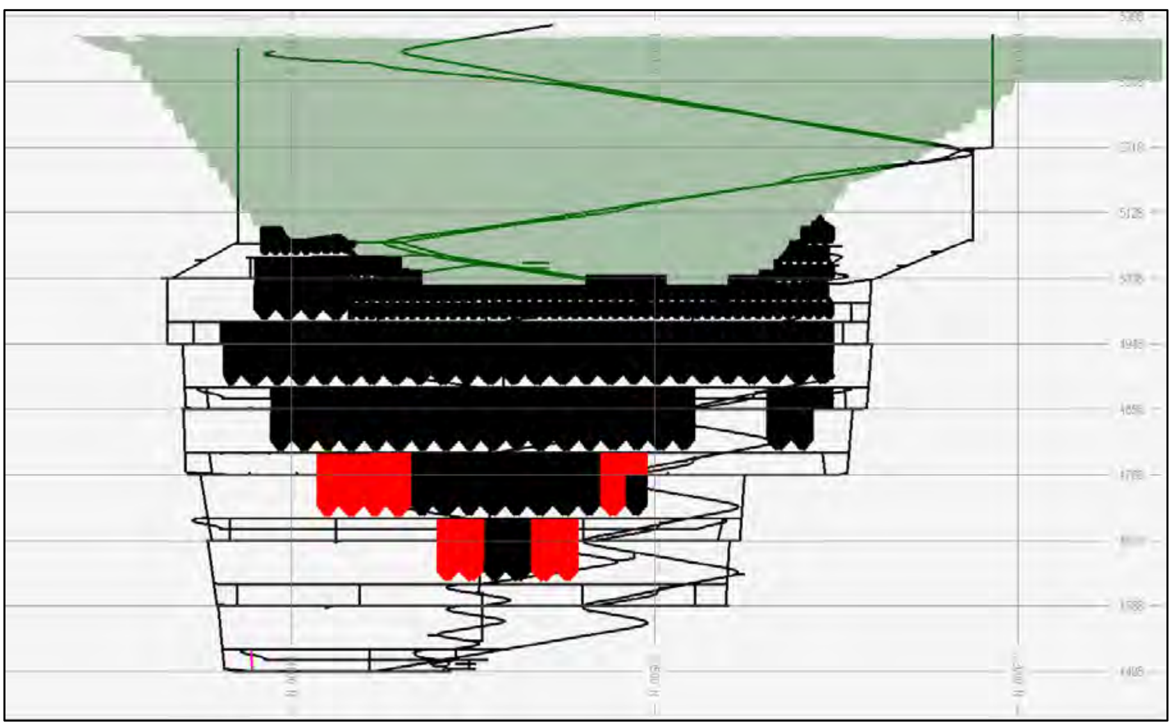


Figure 16-82: End of UG Year 13 (Year 18 of combined schedule)

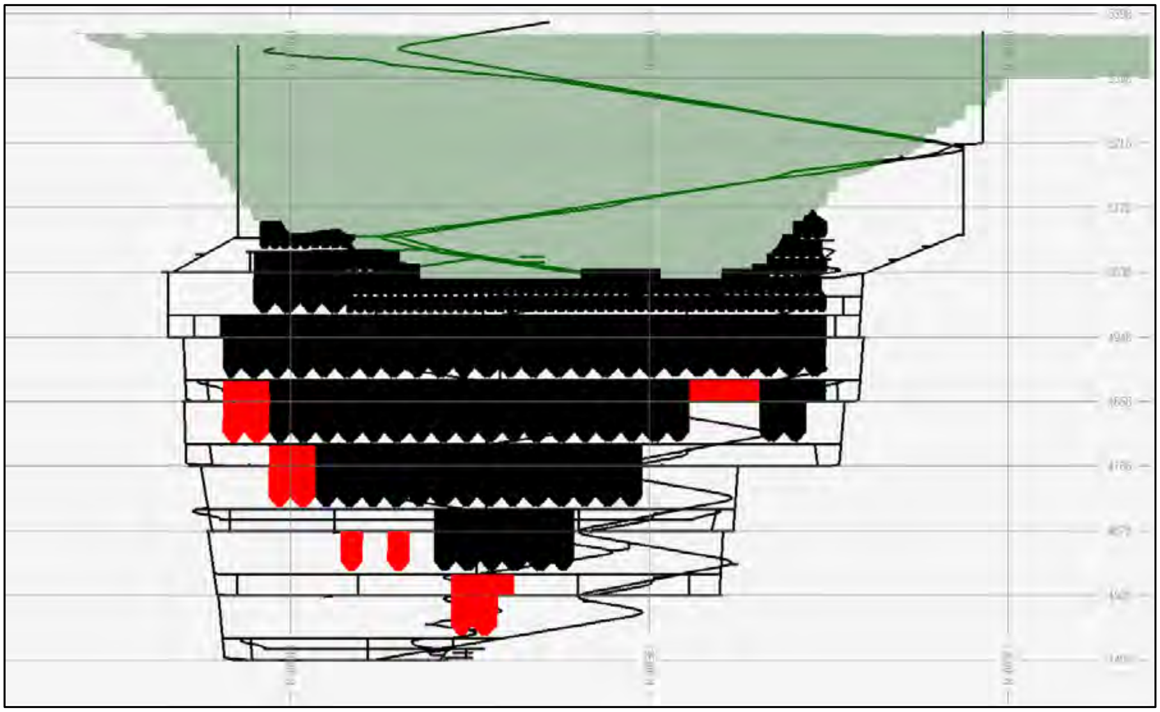


Figure 16-83: End of UG Year 14 (Year 19 of combined schedule)

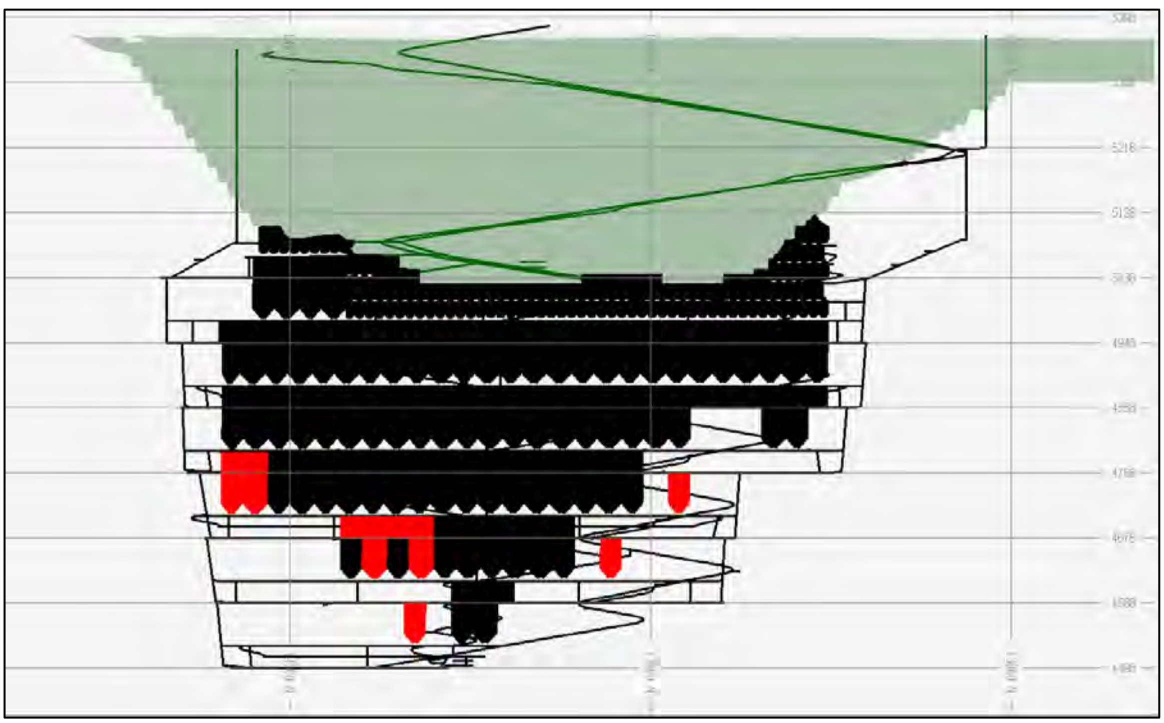


Figure 16-84: End of UG Year 15 (Year 20 of combined schedule)

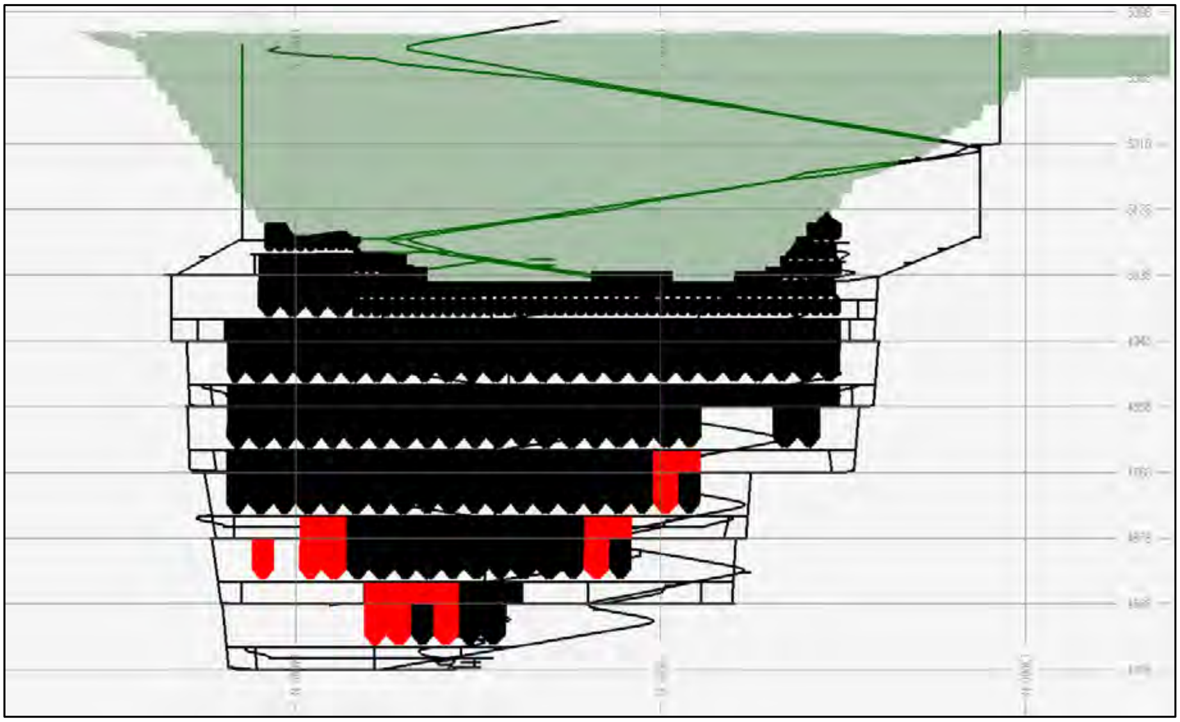
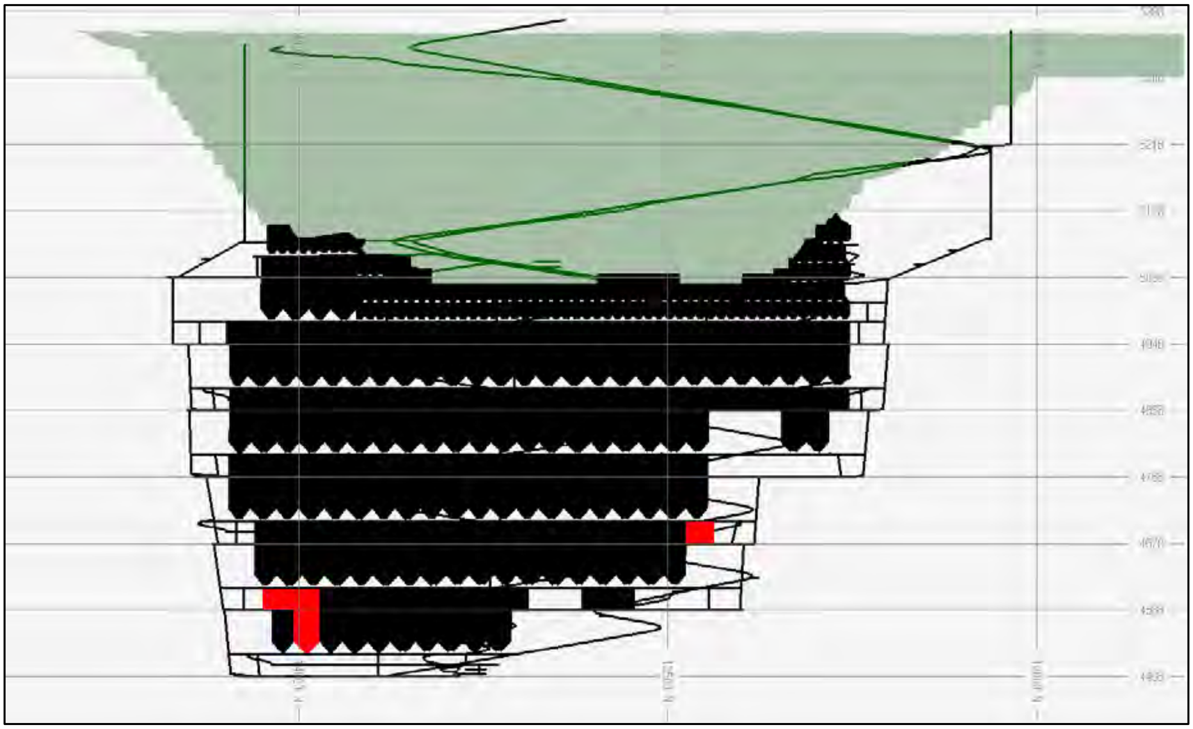


Figure 16-85: End of UG Year 16 (Year 21 of combined schedule)

Figure 16-86: End of UG Year 17 (Year 22 of combined schedule)



17 RECOVERY METHODS

17.1 Process Design

The Troilus process plant design is based on a conventional metallurgical flowsheet to produce a copper-gold flotation concentrate and gold doré from gold gravity concentrates. Although not identical, the current flowsheet design contains many similarities to the process plant that was in production at Troilus between 1996 and 2010. Therefore, the selected flowsheet is considered to be robust and fit for purpose to maximise metal recoveries while minimising capital and operating costs. The flowsheet comprises of primary crushing, milling (SAG and ball mill), rougher, scavenger and cleaner flotation, regrinding, concentrate dewatering, tailings dewatering, conventional dam storage of tailings, and gravity concentration of gold in both the primary milling and regrind circuits, followed by intensive leaching and electrowinning of gravity concentrates to produce gold doré at site.

The key criteria for equipment selection are suitability for duty, reliability, and ease of maintenance. The plant layout is conceived to provide ease of access to all equipment for operating and maintenance requirements.

The key project design criteria for the plant are:

- nominal throughput of 35,000 tpd of ore
- crushing plant availability of 6,570 hours per annum (75%) and process plant (grinding, flotation, dewatering) availability of 8,077 hours per annum (92%) through the use of standby equipment in critical areas, planned maintenance systems and reliable grid power supply
- primary milling circuit has been sized to produce a product size P_{80} of 75µm and the rougher-scavenger concentrate regrind is sized to produce a product size P_{80} of 25µm
- life of mine average copper and gold recoveries to flotation concentrate are estimated to be 90% and 60% respectively, with gold recoveries to doré, via the gravity circuits at 30%
- copper/gold flotation concentrate will be filtered, and then filter cake will be bagged
- gravity concentrates will be intensively leached, and a combination of electrowinning followed by smelting will produce gold doré at site
- final tailing slurry will be thickened and then pumped to the tailing storage facility
- sufficient automation and plant control will be incorporated to minimise the need for continuous operator intervention

The selected process flowsheet has been designed using historical process plant data and metallurgical testwork results from testwork conducted at reputable testwork laboratories in North America. An overview of the testwork completed to date is provided in Chapter 13 of this report.

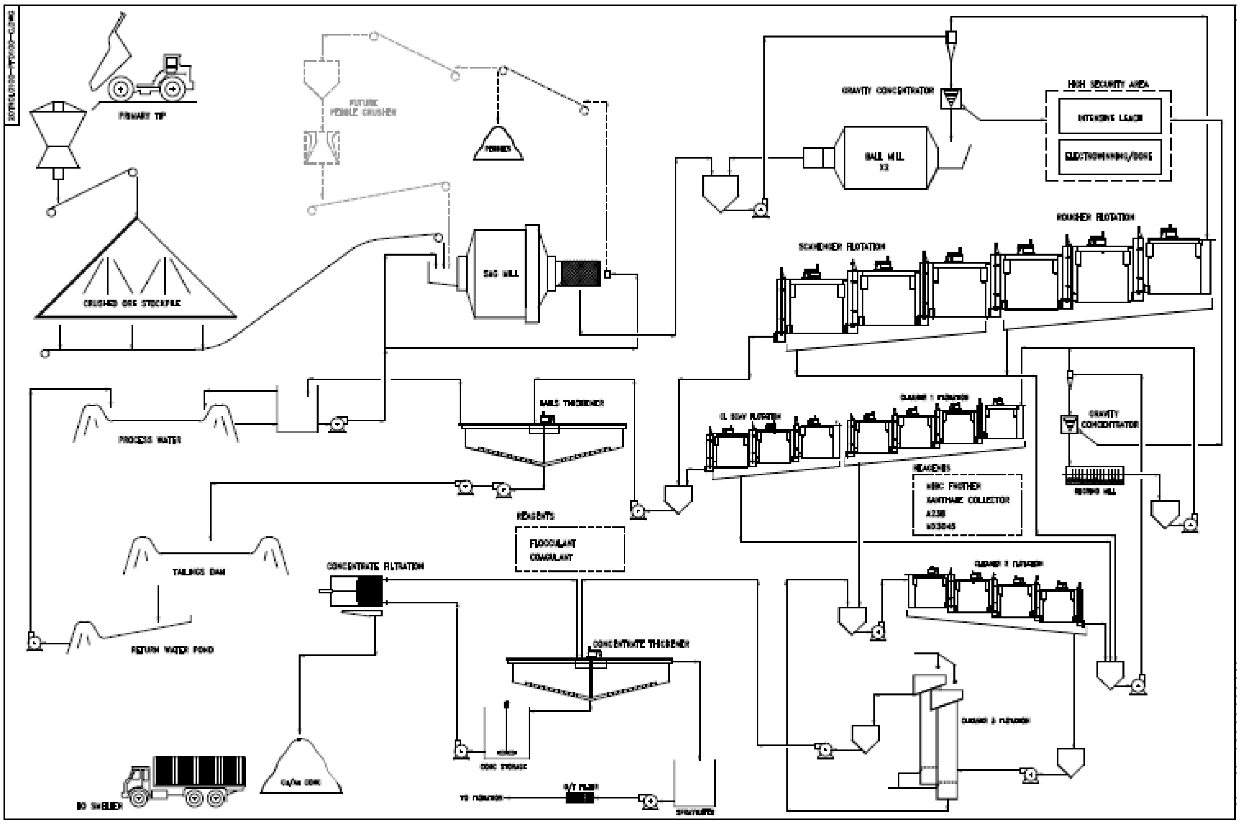
17.1.1 Selected Process Flowsheet

The Troilus process plant has been designed for a throughput of 35,000 dry mtpd. The flowsheet is comprised of the following steps:

- run of mine (ROM) material is dumped directly into the gyratory crusher via 181 tonne capacity haul truck
- crushed ore is stockpiled (18-hour live storage capacity) and two lines of three pan feeders discharge onto the SAG mill feed conveyor
- open circuit SAG milling, with provision for future installation of a pebble crusher to crush SAG trommel oversize material (if required)
- ball mill in closed circuit with hydrocyclones and gravity concentration via Knelson concentrator on the cyclone underflow
- copper-gold rougher and scavenger flotation
- regrinding of the rougher and scavenger concentrate with gravity concentration via Falcon on the regrind cyclone underflow
- copper-gold cleaner flotation, via two stages of conventional cleaning and a third stage of cleaning via column flotation
- concentrate thickening and filtration
- tailings thickening and pumping of thickened tails to the tailing storage facility
- intensive cyanide leaching of the gold gravity concentrates followed by electrowinning and smelting of doré on site

The process flowsheet is summarised in Figure 17-1 below.

Figure 17-1: Process Flowsheet



17.2 Process Plant Description

17.2.1 Process Design Criteria

The salient process design criteria are summarised in Table 17-1 below and were derived from historical metallurgical testwork described in Chapter 13 of this report.

Table 17-1: Process Design Criteria

Parameter	Units	Value	Comments
Plant throughput	dmtpa	12,775,000	
	dmtpd	35,000	
Design basis head grade	% Cu	0.08	
	g/t Au	0.72	
Ore moisture content	%	5 - 6	
Ore density	t/m^3	2.70	Weighted average of waste and ore
Operating Schedule:			
Crushing plant operating time	hrs. pa	6,570	
Process plant operating time (grinding, flotation, dewatering)	Hrs. pa	8,077	
JK Dropweight Index	A x b	40	
Bond Ball Work Index (metric)	kWh/t	14.5	100µm closing size
Bond Abrasion Index	g	0.35	
Primary Grind P_{80}	µm	75	
Rougher/scavenger Res. Time	mins	25.0	
Regrind P_{80}	µm	25	
Cleaner 1 Res. Time	mins	12.5	
Cleaner 2 Res. Time	mins	6.3	
Cleaner 3 Res. Time	mins	8.0	Column flotation
Final Conc. production rate	t/h	11.5	
Copper-Au Conc Grade	% Cu	16.6	LOM Average
	g/t Au	103	LOM Average
Copper recovery	%	90	
Gold recovery (as Flotation Conc.)	%	60	
Gold recovery (as Doré)	%	30	

17.2.2 Primary Crushing – Area 100

Run of mine material will be discharged directly into the primary gyratory crusher by 181 tonne capacity haul trucks at a rate of approximately eleven trucks per hour. Peak delivery rate is assumed to be 2,200 dmtph. The crusher is serviced by a hydraulic rock breaker to handle oversize rocks.

The gyratory crusher can handle rocks with a top size of 800mm and will run with a 170mm open side setting. An apron feeder is used to withdraw crushed mill feed from the surge pocket beneath the crusher onto a short sacrificial conveyor. This conveyor discharges onto the main stockpile feed conveyor. The P_{100} size exiting the gyratory crusher is estimated to be 375mm.

17.2.3 Stockpile – Area 120

The crushed mill feed stockpile provides a live capacity equivalent to roughly 18 hours of plant production, equivalent to 28,471 t live capacity. Mill feed is withdrawn from the stockpile via two lines of three pan feeders. Four pan feeders will operate simultaneously at a peak capacity of 490 dt/h each, with two pan feeders on standby. Alternating feeder use during normal operation is expected to assist with mill feed size distribution control. Each feeder discharges via a lined discharge chute onto the SAG mill feed conveyor via individual dead-boxes.

17.2.4 SAG Mill – Area 200

From the stockpile discharge feeder, mill feed is withdrawn in measured quantities onto the mill feed conveyor. This conveyor discharges via the head chute into the SAG mill feed hopper.

A single 10.4 m diameter x 4.9 m long, twin pinion, semi-autogenous grinding mill equipped with twin 4.5MW variable speed motors is operated in open circuit with a trommel screen on the mill discharge. Slurry exits the mill after passing through the mill discharge grate onto a trommel screen fixed to the mill discharge trunnion. Trommel oversize material is returned to the mill by an integral bucket wheel and rock-box arrangement, with a high-pressure water cannon used to wash material back into the grinding chamber. Trommel screen undersize material gravitates as a slurry to the common mill discharge pump box, from where it is pumped to the ball mill cyclones.

SAG mill slurry spillage is collected in a drive-in sump, and then returned to process by a submersible slurry pump.

The milling area (SAG and ball) is served by a common overhead crane. Relining is achieved using the common relining machine.

SAG mill grinding media is stored in a ball bunker. The bunker is served with a small spillage pump and a ball loading crane and magnet. Balls are added to mill feed at timed intervals via a ball loading chute.

17.2.5 Ball Mill – Area 240

After SAG milling, the particle size is further reduced to a P_{80} of 75 µm by conventional, closed circuit ball milling in two parallel 7.9 m diameter x 10.4 m long overflow discharge ball mills. Each mill is equipped with two 5.6MW motors (i.e.: 11.2MW per mill).

The SAG mill trommel screen undersize is gravity fed to a common ball mill discharge sump, whereupon it is combined with dilution water and ball mill discharge before being pumped to the cyclone classification cluster. The cluster consist of eight cyclones (six operating, two standby).

Cyclone underflow gravitates to a splitter that directs a portion of the stream to a pair of XD70 Knelson concentrators running in parallel. Gravity concentrate is collected and pumped to the high security gold area. Gravity tails discharge by gravity to the feed chute of the ball mill. The cyclone overflow reports to a trash screen for removal of woodchips and other tramp material prior to flotation. The screened cyclone overflow stream gravitates to the flotation circuit via an automatic cross-launder sampler. The stream of woodchips and tramp plastic from the linear screen is dewatered by a woodchip sieve bend before being dumped in a storage area. Spillage contained in the ball mill area is pumped to the common mill discharge sump for re-treatment.

Ball mill grinding media is delivered to the plant in bulk and is stored in the ball mill ball bunker. The ball bunker is serviced by a crawl and electric hoist arrangement, allowing balls to be lifted into a kibble using the ball loading magnet, and tipped into the mill via a ball loading chute.

17.2.6 Rougher Flotation – Area 300

Screened cyclone overflow serves as feed to the rougher flotation section. The rougher/scavenger bank consists of six 300 m^3 cells operating in series, providing 25 minutes of flotation retention time. Flotation air to each cell is supplied by flotation blowers via a low-pressure manifold, and flow is controlled by modulating valves and vent-captor type flow meters. Pulp level is maintained by modulating dart valves.

Rougher/scavenger concentrate is pumped to the regrind mill circuit for additional liberation. Tailings from the final scavenger cell report to a sampling launder, then to a primary sampler, and finally the rougher/scavenger tailings thickener.

Spillage in the rougher section is collected in a common sump and pumped back into the first rougher cell using a submersible spillage pump.

17.2.7 Rougher Concentrate Regrind & Cleaner Flotation – Area 315

Rougher/scavenger concentrate slurry is fed to a regrind cyclone feed pump box and pumped to a set of hydrocyclones. The cyclone underflow gravitates into a Falcon SB750 centrifugal gravity concentrator. Gravity concentrate is collected and pumped to the high security gold area for intensive leaching, electrowinning, and refining. The Falcon concentrator tailing slurry discharges directly into an IsaMill where it is ground to a P_{80} of 25µm, prior to pumping to the head of the cleaner flotation circuit. The regrind cyclone overflow flows by gravity to the cleaner 1 flotation feed box, where it is combined with the regrind mill product.

A spillage pump is used to pump the contents of the area sump back into the regrind mill.

The 1st Cleaner/Scavenger bank consists of five 20 m^3 tank cells in series. Flotation air is supplied from a low-pressure manifold, and pulp level is maintained by modulating dart valves.

The 1st Cleaner concentrate is collected in a common launder and pumped to the 2nd Cleaner feed box. Provision is made for diversion of the concentrate launder from the last two cleaner one cells into a cleaner one scavenger concentrate stream that is pumped to the regrind cyclone feed. Cleaner Scavenger tails slurry gravitates via a sampling launder and sampler to the Rougher Scavenger tails pump box, prior to pumping to the final tails thickener.

The 2nd Cleaner bank consists of three 10 m^3 tank cells in series. Flotation air is supplied from a low-pressure manifold, and is flow controlled by modulating valves.

The 2nd Cleaner concentrate is collected in a common launder and pumped to two parallel 1.8m diameter x 6m high flotation columns. Each column includes a cavitation tube air sparging system, wash water system, and level control. Concentrate collected from the column cells is pumped to the copper concentrate thickener.

The Cleaner area spillage is collected in bermed areas and directed into the cleaner area spillage pump, which pumps back to the 1st Cleaner feed box.

17.2.8 Concentrate Dewatering – Area 350

Final copper concentrate is pumped to the copper concentrate thickener-sampling box and sampler before entering the copper concentrate thickener for dewatering. This thickener is equipped with a rake lift, bed level detection, and bed mass monitoring. Thickener overflow gravitates to the spray water tank for recycling to the flotation circuit, while the thickener underflow is withdrawn from the cone by a centrifugal underflow pump and pumped forward to the copper concentrate storage tank or recycled to the thickener feed if of insufficient density.

The copper/gold concentrate is pumped from the mechanically agitated storage tank to the pressure filter for dewatering. Filtrate from the pressure filter is directed to the concentrate thickener as a recycle stream.

Copper/gold filter cake is discharged from the press via two cake discharge chutes onto the cake transfer belt, which transfers cake to the concentrate storage shed. A front-end loader serves the cake stockpile and loads cake into side-tipping trucks that transport the concentrate to a toll smelter. Trucks are weighed and auger-sampled at the weighbridge prior to dispatch.

Concentrate dewatering area spillage is recovered by pumping back to the concentrate thickener.

17.2.9 Gold Room – Area 400

Gold gravity concentrates from the Knelson (primary grinding circuit) and the Falcon (regrind circuit) are combined ahead of batch intensive leaching in and Acacia leach reactor. The concentrate is prewashed to remove fines, with fines material being pumped back to the ball mill discharge pump box. The deslimed concentrate is leached in the Acacia reactor vessel in batch. After each 8-hour leach cycle, pregnant leach solution is drained from the reactor vessel. The leach residue is pumped back to the ball mill discharge pump box and pregnant leach solution reports to the electrowinning circuit.

The electrowinning circuit plates gold onto a stainless steel wire mesh which is then washed off, collected, dried, and smelted into doré and stored in the gold room vault prior to dispatch.

17.2.10 Tailings Dewatering – Area 500

The rougher scavenger tailing slurry and the 1st cleaner scavenger tailing slurry are pumped to the feed launder of the 55 m diameter high rate thickener. Flocculant and coagulant are added as necessary to give required settling rates and overflow clarity.

Thickener overflow gravitates to the process water tank, while thickener underflow is pumped to the nearby tailings storage facility.

Area spillage is returned to the process by the spillage pump.

17.2.11 Reagents – Area 800

Collector - SIPX

Sodium isopropyl xanthate (SIPX) pellets are delivered to site in 1-tonne bulk bags and stored in the reagent storage area. Bags are added to the mixing tank via the reagent area hoist and collector loading chute. Collector is mixed to 10% solution strength within the tank, and then transferred to the storage tank, ready for distribution.

From the storage tank, collector solution is continuously pumped to the collector head tank, which in turn overflows back to the mixing tank. Peristaltic hose pumps meter collector solution to several addition points throughout the plant.

Reagent spillage is pumped to the tailings tank for disposal on the tailings dam. The reagent area is served by a safety shower.

Collector – Aero 3477

Liquid collector Aero 3477 is delivered to site in 1 m^3 totes. As delivered (full strength) Aero 3477 is pumped directly to the dosing points by dedicated peristaltic pumps.

Frother – MIBC

Liquid methyl isobutyl carbinol (MIBC) is delivered to site in bulk tankers and transferred to an on-site storage tank. As delivered (full strength), MIBC is pumped from the storage tank directly to the dosing points by dedicated peristaltic pumps.

Flocculent – Magnafloc 10

Flocculant powder is delivered to site in 500-kg bags and stored in the reagent storage area. Bags are lifted by the reagent area crane and added to the flocculant powder hopper. Powder is withdrawn by the flocculant screw feeder and blown through a venturi to a wetting head located on top of the mechanically agitated mixing tank.

From the mixing tank, mixed flocculant can be fed forward to the storage tanks or recycled back into the mixing tank to aid mixing. Once mixed, the flocculant should be left for several hours to hydrate. A storage tank provides sufficient volume for storage of flocculant while the mixed batch hydrates in the mixing tank. From the storage tank, flocculant is pumped directly to the tailings and concentrate thickeners.

pH Control – Lime

Hydrated lime is delivered to site by truck and transferred using a vacuum system to the lime silo for bulk storage on site. An automated mixing system meters hydrated lime from the silo and a corresponding volume of water to produce a lime slurry at the required concentration. This lime solution is prepared in batches at a solution strength of 10% then transferred to a lime slurry storage tank. From the storage tank, lime slurry is pumped via a ring main system to various addition points around the plant.

17.2.12 Services – Area 900

Process water is stored in an insulated 1,200 m^3 tank and is distributed to the plant by the process water pumps.

Clean water is piped into the plant from wells and stored in the plant clean water tank. From the storage tank water is pumped around the plant for use as reagent mixing water, slurry pump gland seal water, and, when required, for mill lubrication system cooling.

Plant instrument air is provided by two compressors. Air quality is maintained by a filtering system and a refrigeration drier. Two large air receivers are provided for compressed and instrument air lines, to allow for surges in demand.

Low-pressure air is supplied to the flotation plant by four separate blowers. The blowers are fixed speed, with manifold pressure controlled by a modulating valve on an exhaust line. Separate pressure control for roughers and cleaners is assumed.

17.3 Energy and Water Requirements

As part of the preliminary design and costing exercise, a list of mechanical equipment has been prepared, including the estimated power requirements. A summary of connected and drawn power is given by area in Table 17-2 below.

Table 17-2: Connected Power

PFD Index		Connected kW	% Power Utilization	Daily kWh Consumed
100	Primary Tip & Gyratory Crushing	1188	66%	18,778
120	Coarse Ore Stockpile	352	81%	6,836
200	SAG Milling Circuit	9,361	82%	185,220
240	Ball Milling Circuit	24,966	83%	496,759
300	Rougher Flotation	2,458	81%	47,986
315	Cleaner Flotation	1,807	82%	35,584
350	Copper Conc. Dewatering	178	78%	3,347
400	Gold Room	520	72%	8,928
500	Final Tailings Dewatering	697	81%	13,621
600	Reagents	149	81%	2,915
700	Services (Water, Air)	1256	78%	23,613
TOTAL		**42,932**	**82%**	**843,588**

NOTE: Connected Kw does not include standby equipment

Raw and process water is consumed within the plant, and process water is recovered from product slurries within the two thickeners. Tailing slurry is pumped to the tailing storage facility, and the water within that slurry represents the overall plant water consumption, less any return water pumped back to the plant. The preliminary mass and water balance for the proposed flowsheet indicates that between 20,000 and 22,000 m3 of raw water would be required each day.

18 PROJECT INFRASTRUCTURE

18.1 Overall Site

The overall site plan is shown in Figure 18-1 and includes major facilities of the Troilus Project. This includes:

- 87 Zone pit – only one phase in plan with the backfilled outline shown
- J Zone pit – final phase of three planned for mining
- SW Zone pit – final phase of two planned for mining
- Primary crushing and conveying
- Process plant location
- Overburden and Waste Rock Storage facilities
- No Name Creek diversion
- Tailings Facility – at end of mine life with buttress waste storage facility

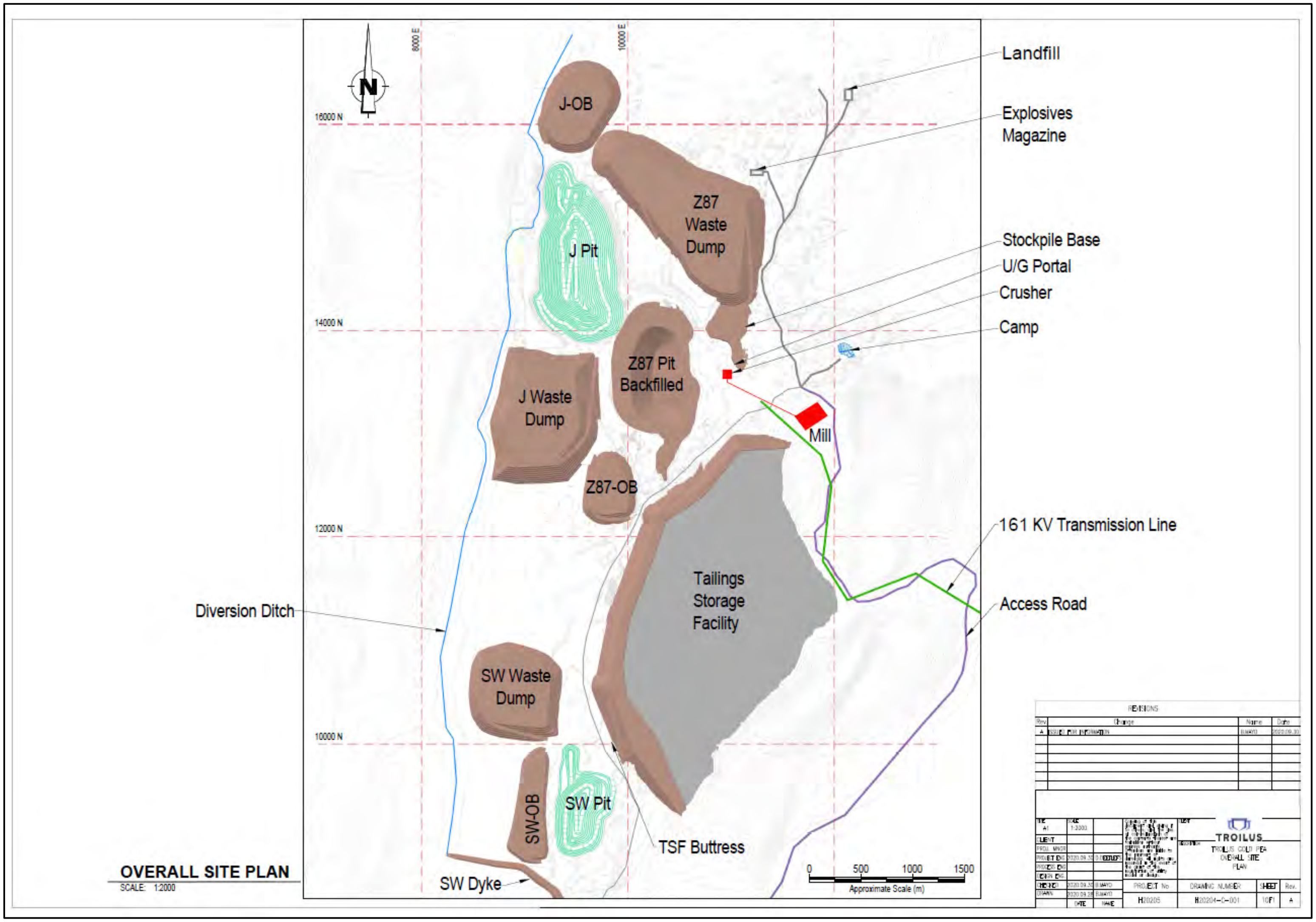
Access to the site will be provided by a new access road to the south and east of the tailings facility.

Grid power will be provided by the existing high voltage line to the current transformer substation.

Raw water will be provided by an existing facility located in the lake north of the site and reclaim of water from the tailings facility

The existing No Name Creek diversion will be extended and realigned to provide access to waste storage facilities.

Figure 18-1: Troilus Project – Site Infrastructure



18.2 Waste Storage Facilities

Waste rock and overburden will be stored in separate locations. The overburden material will be used for later resloping of the waste facilities. These storage facilities will be constructed in 10 metre lifts. Waste facilities will be actively reclaimed as they are constructed with dozers resloping to 26.5 degrees. This is to allow revegetation to occur as soon as possible.

The 87 waste dump area will be reactivated for storage which will allow to place material on top on the free dumping sites to be coherent with the successful reclamation already observed at the Troilus Site. Some of the J zone past waste dumps will be rehandled to accommodate the larger J zone pit.

A portion of the 87 Pit will be backfilled with waste material from the J zone phases. The entire pit is not backfilled due to timing of the various mining activities. The final interior portion will be resloped for reclaim purposes as well.

Overburden volumes use a 1.20 swell factor while waste rock uses a 1.30 swell factor for volume determination.

Waste rock volumes estimated are shown in Table 18-1.

Table 18-1: Waste Storage Facility Volumes

Waste Storage Facility	Storage Capacity (Mm^3)	Top Lift Elevation (masl)
Stockpile Base	1.1	5390
TSF Buttress	39.0	5440
J Overburden	14.5	5400
Z87 Overburden	6.4	5400
Z87-WD	49.7	5440
J-WD	41.0	5420
Z87 Backfill	105.1	5365
SW Dyke	0.8	5395
SW Overburden	4.2	5400
SW-WD	30.3	5430
Total	**292.1**	

18.3 Tailings Storage Facilities

Tailings will be stored in the existing facility location. This facility will be expanded annually to accommodate the expected process tonnage. The facility has sufficient capacity to accommodate this tonnage.

The material will be stored as a thickened tails with water reclaimed from the facility to offset process freshwater requirements.

The facility will be expanded in a center line construction manner. Material for the dam will be provided by the mine which will bring additional material to buttress the facility.

18.4 Camps and Accommodation

Small scale camp facilities exist on site for use in exploration at this time. The previous Troilus camp site is a flat pad with all the water and sewer connections ready to be reconnected.

Camp requirements are for 350 persons initially rising to 425 persons as the underground mine is established. As the underground mine reaches steady state production levels, the open pit manpower levels will drop to that required to maintain the process plant at 35,000 tpd until the open pits are complete. Then underground mining personnel together G&A and process will be present on site.

The camp facilities are included as part of a quotation provided by a local vendor to supply all camp facilities and catering for the project life. The facilities will have accommodations, catering, lounges, and a fitness centre for Troilus personnel.

A nominal cost to setup the camp is included in the capital estimate, but ongoing costs are covered in the cost per person in camp under the G&A costs.

18.5 Power and Electrical

Existing electrical infrastructure includes the Hydro Quebec 161 MVA line to site. At site there are two 25MVA transformers in the current substation. Diesel backup power is also at the substation.

This existing electrical infrastructure is sufficient for the PEA outlined requirements.

The high voltage line will need to have 4-5 poles relocated as the tailings facility expands. This cost would be borne by Hydro Quebec and would be completed prior to plant commissioning.

18.6 Support Buildings

The following support buildings are required for the project and costs have been allocated for them:

- Mine Truck Shop and Warehouse – 10 bay shop with tire bay, wash bay and electrical bay
- Explosives Storage –old facility to the north of the 87 pit will be reactivated
- Fuel Storage – sufficient capacity for 3 days supply will be constructed
- Administration Building
- Assay Laboratory – separate assay facility will be constructed to handle mine and plant samples
- Security Gate and Weigh Scale

18.7 Water Supply

Process water will be sourced from two locations:

- tailings pond water reclaim

- fresh water from lake to north of 87 zone

The previous operation installed a water pipeline and lake inlet north of the current mine. This system is available for use upon the project start-up for process water.

Potable water was sourced to the south east of the property adjacent to the past camp facilities. This system will be recommissioned for project use.

18.8 Water Management

Water treatment facilities current exist at the tailings facility and J zone pit. The equipment at the J Zone pit is not required currently but would be expanded during operation to accommodate anticipated water pumping volumes. This expansion capital is included in the capital cost estimate.

Pit pumping requirements are estimated at 12,000 m^3 per day with a seasonal peak of 15,000 m^3 per day. Open pit and underground costing and design will accommodate this quantity.

When Troilus operated previously, No Name Creek was diverted around the 87 and J zone pits with a ditch. With the expanded footprint for the PEA, the diversion ditch will be extended and realigned to the western side of the valley. This is to avoid the new SW zone pit and waste storage facility as well as the enlarged waste dumps for the 87 and J zone areas.

Due to elevation differences, the ditch will start further to the south of the present start of the ditch. It will also include a small dike to direct the water to the proper elevation. This dyke will create additional habitat for potential fisheries.

19 MARKET STUDIES AND CONTRACTS

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, dore gold will be produced from gravity separation in the process plant which will be sold into a different market.

Increased demand for copper in the Asian markets has stimulated the expansion of processing capacities for copper raw materials in Asia, and rationalized reduction and/or elimination of similar existing processing capacities elsewhere in the international market. The balancing of supply and demand is expected to continue where newly created processing capacity should absorb much of the new copper concentrate production capacity that will be realized.

The quantity and estimated value of the mine's product will allow movement of product to multiple regions, therefore, the regions and consumers providing the least commercial risk and the optimum return to the Project should be considered. Considering the higher gold content of the concentrate, those locations providing payables for gold need to be properly considered.

19.1 Metal Pricing

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

- Gold - $US1,475 /oz
- Copper - $US3.00 /lb
- Silver - $US20.00 /oz

The metal price history for spot, two-year, three-year, five-year and 10-year are noted in Table 19-1

Table 19-1: Metal Price History

Metal	Unit	Spot Price – August 21,2020	2 Year	3 Year	5 Year	10 Year
Gold	$US/oz	$1,935	$1,462	$1,406	$1,332	$1,383
Copper	$US/lb	$2.98	$2.69	$2.81	$2.63	$3.02
Silver	$US/oz	$26.71	$16.51	$16.52	$16.58	$21.19

The prices shown in Table 19-1 formed the basis for the price determination of pit designs and economic analysis. The periodicity of copper and silver favoured the longer price outlook where gold and considerations of market economies and their projects favoured a closer price period.

19.2 General Considerations

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 16.6% grade, the higher gold grade of 103 g/t will make it attractive to various worldwide smelters.

Proceeding forward, AGP would recommend the mine focus upon selective smelting and refining complexes that currently process copper concentrates in Europe, or along the Pacific Rim, to maximize the gold value obtained from the concentrate sales.

Handling considerations will be slightly different than normal concentrate in bulk as this is bagged to minimize the losses of gold during transportation.

Logistics must be further examined; however, freight costs should not greatly influence the value of the concentrate. The mine is within reasonable access to port or rail facilities to allow transport of the product.

Normal deviations in moisture content and the methods established to sample and determine the settlement dry weight must be closely examined and controlled. Moisture samples should be taken when product is weighed and sampled for assays. Care must be taken to immediately seal the moisture samples and follow the established procedures for drying and determination of dry weight. Sampling for assay determination should be examined but will likely follow normal procedures. Samples are taken from the bags when departing the mine area, and possibly again upon loading of the carrying vessel. Here, a frequently calibrated static scale will be utilized before the trailers or cars are discharged at the storage area, prior to loading of the vessel. The trailers or cars must be recorded for tare weight as well as total weight. The storage area for loading the carrying vessels must be very secure.

Assaying, exchange of assay results, and the splitting limits for determination of settlement results must be professionally managed. The use of bagged concentrate should avoid unnecessary losses in handling and transport of the precious metals.

19.3 Terms and Conditions Discussion

19.3.1 Accountable Metals

For the purposes of this study, the assumption was made that copper concentrate would be comprised of the following accountable elements:

- Copper
- Gold – portion not extracted via gravity concentrator
- Silver – all the silver value is being considered in the concentrate and none in the dore

19.3.2 Smelting and Refining Charges

The proposed smelting and refining terms for each product are consistent with normal market trends. Indicative smelter terms were provided to Troilus for the PEA 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. 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.

Table 19-2 shows the terms applied to the study LOM average copper concentrate grades. These terms are considered reasonable for the purposes of the PEA study.

Table 19-2: Smelting and Refining Terms – LOM Average

Term	Unit	Copper	Gold	Silver
Cu, Au Minimum Deduction	%, g/dmt	1.2%	0.0 g/dmt	30.0 g/dmt
Base Smelting Charge	$US/dmt	62.00	-	
Cu Refining Charge	$US/lb payable	0.062	-	
Payable	%	100.0	96.5	90.0
Refining Charge	$US/oz	-	5.00	0.50
Concentrate Grade (LOM average)	%, g/t	16.6%	103 gpt	94 gpt
Concentrate Moisture	%	8.0	-	

It should be noted that the terms provided for the copper concentrate considered a sliding scale for minimum deductions and copper payable based on the final copper concentrate grade. The variable terms are shown in Table 19-3.

Table 19-3: Copper Concentrate Variable Terms

Concentrate Grade (Cu%)	Copper Payable (%)	Copper Deduction (%Cu units)
12% - 14%	100%	1.4 units
14% - 16%	100%	1.3 units
16% - 18%	100%	1.2 units
18% - 20%	100%	1.1 units
20% - 22%	96.5%	1.0 units

For the gold dore produced from the gravity concentrator, the following terms were applied:

- Gold Payable – 99%
- Dore Selling Cost - $12 US/oz

Dore is shipped to major refineries and terms and conditions would be consistent with standard industry practices.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Background

The Troilus 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 Troilus site has currently two environmental statuses: exploration and closed(reclaimed) sites. The site has been reclaimed from the end of the previous operation, from 2011 to now. The waste piles and tailings pond have been revegetated The remaining work for closure is removing the pumps from the tailings and having the water flow naturally via a canal. The exploration status relates to the drilling and finding new resources for an eventual new operation.

In November 2019, the Company submitted an environmental impact study 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 Troilus property. The Company engaged in community consultations with impacted families on the Troilus 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.

20.2 Baseline Studies

Baseline studies were conducted prior to the exploitation of the Mine in 1997 and due to the elapsed time, new baseline studies were undertaken by various consultants in 2019, to consider the following environmental and social aspects:

- site hydrology
- surface and groundwater quality
- climate and air quality
- vegetation and wetlands
- fish and associated habitat
- terrestrial and avian wildlife
- archeological values
- land use and resources
- socio-economics

The baseline studies have and will continue to focus on a description of existing conditions, considering that the site has already been impacted by the operation of a mine for about 12 years, then has been partially restored.

20.3 Environmental Issues

No significant or material issue are foreseen with advancing the project in its proposed configuration.

20.4 Closure Plan

The site is currently in a closed(reclaimed) state, along with areas being under exploration.

As the project advances through various stages of study, a closure plan will have to be made for the future Project incorporating updated practices and regulations. This would replace the current closure plan that is currently in effect.

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

20.5 Permitting

Under the James Bay and Northern Quebec Agreement (JBNQA), an advisory committee was established for projects in the Eeyou Istchee region south of 55°, the James Bay Advisory Committee on the Environment (JBACE). There are four members each from Quebec, Canada, and the Cree Regional Authority plus one person representing hunting, fishing, and trapping. The JBACE created two additional committees:

1. Comité d'évaluation des répercussions sur l'environnement et le milieu social (COMEV)
2. Quebec/Cree/Canada bureau/agency for assessing project descriptions and preparing guidelines for an Environmental and Social Impact Assessment (ESIA)
3. Comité d'examen des répercussions sur l'environnement et le milieu social (COMEX)
4. Quebec/Cree bureau for reviewing regional projects.

The Project review process will be composed of five steps:

- the proponent prepares and submits a detailed Project Description to COMEV
- COMEV assesses the Project, its potential impact and prepares guidelines for the Project ESIA
- the proponent prepares an ESIA and submits it to COMEX
- COMEX, with input from the Cree people and Quebec public, reviews the ESIA
- the COMEX administrator renders a decision

Following a successful review process, and assurance of compliance with Quebec Law (including Directive 019), submissions will be made to the Cree Authority and to the Government of Quebec for certificates of approval and permits.

On August 28, 2019, the Impact Assessment Act, the Canadian Energy Regulator Act, and the Canadian Navigable Waters Act came into effect. The Impact Assessment Act created the new Impact Assessment Agency of Canada and repealed the Canadian Environmental Assessment Act, 2012. The Physical Activities Regulations gives the new threshold of 5,000 tpd of extraction as the trigger to have a federal environmental impact assessment.

The Project will exceed this production rate threshold and a federal environmental assessment (EA) will be required. Here are the steps required for the federal ESIA:

- a project description review is submitted to the agency
- determination if an environmental assessment is required
- the company must file an environmental assessment
- agency begins analysis of the environmental assessment
- environmental assessment report is written by the agency
- environmental assessment decision provided by the ministry
- approvals from federal departments and follow-up

20.6 Considerations of Social and Community Impacts

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 employs a fulltime Cree community liaison officer, 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 Quebec 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.

20.7 Discussion on Risks to Mineral Resources and Mineral Reserves

No known environmental issues have been identified at the site that would materially affect the current mine, design, or scope of the needed environmental permits.

The diversion of the unnamed creek as proposed in the PEA will have to examined, as this will be the major environmental item for the Project.

The most substantive potential impacts of projects are generally associated with the long-term management of waste rock, tailings, mine water and process water and their downstream effects on water and fish habitat. As the project advances through the various stages of study, the application of appropriate engineering design, project planning, and implementation of responsible production and environmental management plans will mitigate any significant environmental effects.

The fact that the tailings area and waste piles have been on site since 1997 from the former mine with no significant environmental effects indicates that the risk of having issues with the same orebody is expected to be very low.

21 CAPITAL AND OPERATING COSTS

21.1 Summary

The initial and life of mine capital cost estimate is summarized in Table 21-1. All costs are expressed in Canadian Dollars (CDN) unless otherwise stated and are based on 2020 H1 2020 pricing.

Table 21-1: Troilus Project Capital Cost Estimate

Area	Initial Capital (M$CDN)	Sustaining Capital (M$CDN)	Total Capital (M$CDN)
Open Pit – Prestrip (capitalized)	94.7	-	94.7
Open Pit - Capital	8.4	6.7	15.2
Open Pit Mining - Subtotal	103.1	6.7	109.9
Underground Mining	-	559.7	559.7
Processing	191.3	25.5	216.8
Infrastructure	42.1	22.2	64.3
Environmental	-	25.0	25.0
Indirects	64.4	8.2	72.6
Contingency	48.6	35.2	83.8
Total	**449.5**	**682.6**	**1,132.1**

The life of mine operating cost estimate summary is shown in Table 21-2.

Table 21-2: Troilus Project Operating Cost Estimate (CDN)

	Units	Open Pit Only (Year 1 – 5)	Open Pit & U/G (Year 1 – 14)	U/G Only (Year 15 – 22)	Life of Mine (Year 1-22)
Open Pit Mining	$/t moved	2.73	2.70	-	2.70
	$/t mill feed	15.62	12.62	-	12.62
Underground Mining	$/t mill feed	-	19.54	19.26	19.38
Processing	$/t mill feed	6.74	6.74	6.74	6.74
G&A	$/t mill feed	1.48	1.60	4.19	1.92
Concentrate Trucking	$/t mill feed	0.26	0.32	0.26	0.32
Total Operating Cost	**$/t mill feed**	**24.10**	**22.05**	**30.45**	**23.08**

The study was completed in Canadian dollars but has been shown below in United States Dollars (USD) for easy comparison to other projects nominated in that currency. The project exchange rate of 1.35:1 (CDN:USD) is used to convert to United States Dollars. The capital costs are shown in Table 21-3 and the operating costs in Table 21-4.

Table 21-3: Troilus Project Capital Cost Estimate (USD)

Area	Initial Capital (M$USD)	Sustaining Capital (M$USD)	Total Capital (M$USD)
Open Pit – Prestrip (capitalized)	70.2	-	70.2
Open Pit - Capital	6.2	5.0	11.2
Open Pit Mining - Subtotal	76.4	5.0	81.4
Underground Mining	-	414.6	414.6
Processing	141.7	18.9	160.6
Infrastructure	31.2	16.5	47.6
Environmental	-	18.5	18.5
Indirects	47.8	6.0	53.8
Contingency	36.0	26.1	62.1
Total	**333.0**	**505.6**	**838.6**

Table 21-4: Troilus Project Operating Cost Estimate (USD)

	Units	Open Pit Only (Year 1 – 5)	Open Pit & U/G (Year 1 – 14)	U/G Only (Year 15 – 22)	Life of Mine (Year 1-22)
Open Pit Mining	$USD/t moved	2.03	2.00	-	2.00
	$USD/t mill feed	11.57	9.35	-	9.35
Underground Mining	$USD/t mill feed	-	14.47	14.26	14.36
Processing	$USD/t mill feed	4.99	4.99	4.99	4.99
G&A	$USD/t mill feed	1.09	1.19	3.10	1.42
Concentrate Trucking	$USD/t mill feed	0.20	0.24	0.20	0.23
Total Operating Cost	**$USD/t mill feed**	**17.85**	**16.33**	**22.55**	**17.10**

21.2 Capital Cost

21.2.1 Summary

The capital costs for the Troilus project in the various areas and separated by Initial Capital and Sustaining Capital is shown in Table 21-5. All costs are expressed in Canadian Dollars (CDN) unless otherwise stated and are based on 2020 H1 2020 pricing.

Table 21-5: Capital Cost Estimate

Area	Initial Capital (M$CDN)	Sustaining Capital (M$CDN)	Total Capital (M$CDN)
Open Pit – Prestrip (capitalized)	94.7	-	94.7
Open Pit - Capital	8.4	6.7	15.2
Open Pit Mining - Subtotal	103.1	6.7	109.9
Underground Mining	-	559.7	559.7
Processing	191.3	25.5	216.8
Infrastructure	42.1	22.2	64.3
Environmental	-	25.0	25.0
Indirects	64.4	8.2	72.6
Contingency	48.6	35.2	83.8
Total	**449.5**	**682.6**	**1,132.1**

21.2.2 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 vendors, no down payment was required if the full fleet was purchased from them. The costs are then distributed over the operating costs discussed later in Section 21.3.2.

Equipment pricing was based on quotations from local vendors predominantly and 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 ($CDN)

Equipment	Unit	Capacity	Capital Cost ($CDN)	Full Finance Cost ($CDN)
Production Drill	mm	200	3,111,000	3,319,000
Hydraulic Excavator	m^3	22	9,541,000	10,180,000
Production Loader	m^3	23	8,419,000	9,541,000
Haulage Truck	t	181	4,555,000	4,860,000
Crusher Loader	m^3	13	2,398,000	2,559,000
Tracked Dozer	kW	455	1,609,000	1,717,000
Grader	kW	163	423,000	451,000

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 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, 28 units are necessary to maintain mine production. This happens from Year 1 onwards. The maximum hours per truck/per year are set at 6,000. There are periods where the maximum hours per unit are below what the maximum possible can be. 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 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 three 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 -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
Production Drill	25,000	4	5						1	6						
Production Loader	35,000	1	1													
Hydraulic Shovel	60,000	2	2								1					
Haulage Truck	60,000	9	14	5												
Crusher Loader	25,000		1								1					
Tracked Dozer	35,000	4	2							4	2					
Grader	20,000	3								3						

Table 21-8: Equipment Fleet Size

Equipment	Unit Life (hrs)	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
Production Drill	25,000	4	9	9	9	9	9	9	6	7	7	7	7	7	7	7
Production Loader	35,000	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Hydraulic Shovel	60,000	2	4	4	4	4	4	4	4	4	4	3	3	2	2	2
Haulage Truck	60,000	9	23	28	28	28	28	28	28	28	28	28	28	28	19	19
Crusher Loader	25,000		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
Grader	20,000	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 ($CDN)

Equipment	Preproduction Year -2, -1 ($)	Sustaining ($CDN)	Total ($CDN)
Pre-Production Stripping	94,700,600	-	94,700,600
Mining Equipment			
Major Equipment	3,238,000	504,000	3,742,000
Support Equipment	1,760,000	885,000	2,645,000
Subtotal – Mining Equipment	**4,998,000**	**1,389,000**	**6,387,000**
Miscellaneous Mine Capital			
Engineering Office Equipment	1,200,000	-	1,200,000
Dispatch System	1,000,000	-	1,000,000
Dewatering System	1,239,200	5,346,800	6,586,000
Subtotal – Miscellaneous Mine Capital	**3,439,200**	**5,346,800**	**8,786,000**
Total Mine Capital	**103,137,800**	**6,735,800**	**109,873,600**

Pre-Production Stripping

The mine is scheduled to initiate mining in Year -1. 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 39.5 million tonnes will be mined in this time period and the costs are being attributed to capital. This is expected to cost $94.7 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 mining equipment did not finance such as spare boxes, or buckets is included in this category. It also includes the blasting truck, and pump truck, two specialized units as well as the ambulance, fire truck and associated rescue equipment, a 35 ton rough terrain crane and a 100 ton 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.

Underground Mining

The underground mine capital estimate was split into two phases, project capital phase and sustaining capital thereafter. The project capital phase was assumed to terminate at the end of 2nd quarter Year

9 as significant mill feed production levels are achieved. A 12% contingency was applied to the capital estimate. The estimate comprises four main areas of expenditure:

- Capital Development: Defined as all lateral access and raise development in waste. Stope development was classified as operating development.
- Operating costs incurred during the project capital phase to 2nd quarter Year 9 were transferred into the capital estimate.
- Mobile Equipment: Mobile equipment, including a 5% allowance for initial equipment spares, will be leased. Leasing terms of 20% down payment followed by a five year payment period at the rate of 1.78% per month were applied against all the owners new and replacement mobile equipment acquisitions. Equipment pricing for major equipment types was based on budget quotations. The 20% down payment was classified as capital expenditure while the five year leasing payments were classified as operating expenditure. A mid-life rebuild equivalent to 50% of the purchase price was assumed for all items of equipment in order to extend the useful life. The entire rebuild costs were classified as sustaining capital.
- Mine infrastructure: Preliminary engineering design and cost estimation or vendor budget quotations were applied for the Rail-Veyor system, dewatering, ventilation fans and air heating, power distribution system, communication system, workshop, and fuel storage. These areas represent 71% of the infrastructure capital estimate. Allowances were made by AGP for the following remaining areas:
    - development contractor mobilisation/demobilisation
    - portals
    - UG Lighting
    - service water system
    - tips and grizzlies
    - ventilation doors, seals, and regulators
    - temporary portable 16-man refuges
    - mobile compressors
    - U/G explosive storage
    - permanent refuge stations/lunchroom/shift boss's office
    - stench gas system
    - handheld drills
    - caps lamps
    - mine rescue equipment
    - miscellaneous facilities
    - infrastructure EPCM and indirects

A summary of the underground mine capital estimate is provided in Table 21-10. Additional detail of expenditure during the project capital and sustaining capital periods are shown in Figure 21-1 and Figure 21-2 respectively.

Table 21-10: Summary of the Underground Mine Capital Estimate (CDN$ 000's)

	Project Capital	Sustaining Capital
Capital Development	157,613	146,078
Operating Cost Transferred to Capital	42,834	-
Mobile Equipment	10,787	40,294
Mine Infrastructure	112,847	49,230
Contingency	-	-
Total	**324,081**	**235,601**
Unit Cost CDN $/tonne	7.66	5.57

Figure 21-1: Project Capital Expenditure – from start of Underground Mining

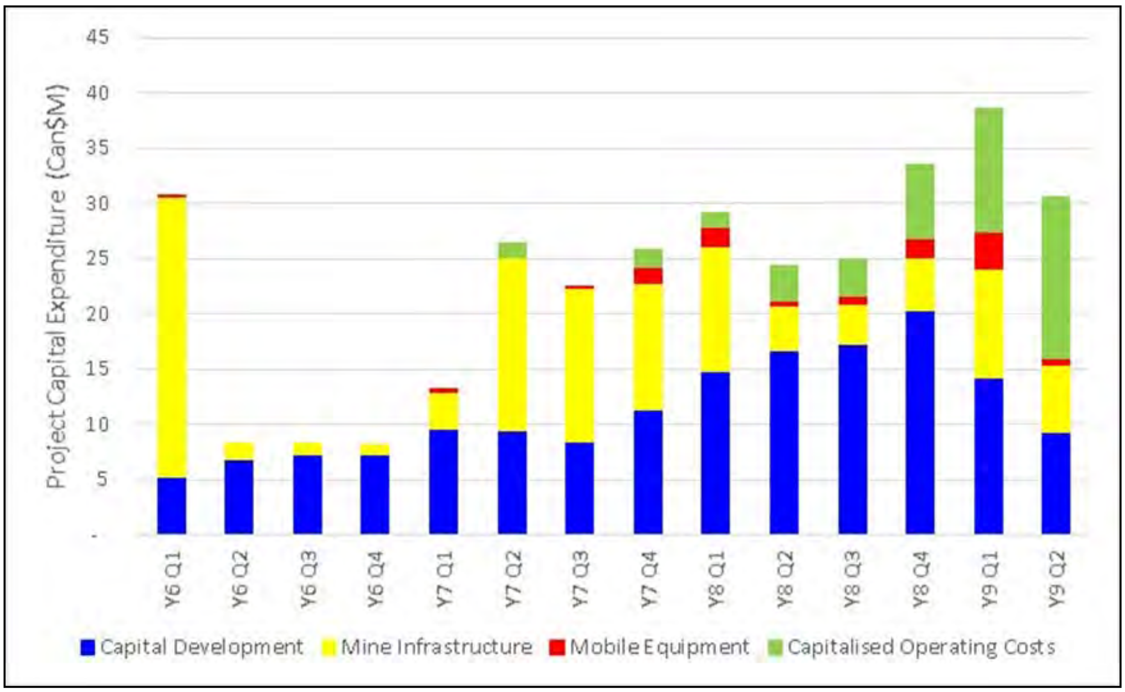
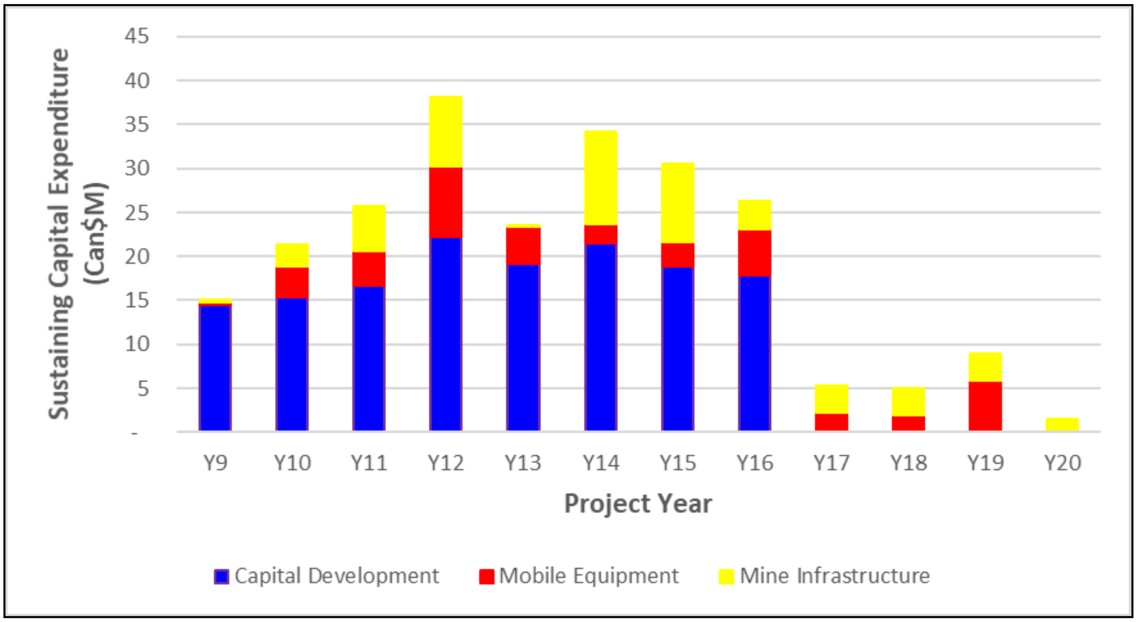


Figure 21-2: Sustaining Capital Expenditure – from start of Underground Mining



21.2.3 Process Capital Costs

The capital costs for the process plant are based on the facilities described in Chapter 17 of this report and prepared by AGP. The purpose of the capital cost estimate is to provide substantiated costs suitable for use within a preliminary assessment of Project economics. Costs have been built up using the equipment lists prepared for the project, together with preliminary layouts and quantities. Costs for mechanical equipment were either based on Vendor budget quotations (for major items) or AGP database costs (taken from other recent similar AGP studies).

The following capital cost estimate is expressed in CDN Dollars unless otherwise stated and is based on Q2 2020 pricing, deemed to have an overall accuracy of +/-35%. Direct capital costs are provided in Table 21-11.

Table 21-11: Process Direct Capital Costs ($CDN)

Area	Total Cost, $CDN
100 – Primary Gyratory Crushing	$29,283,600
120 – Coarse Ore Storage	$8,955,200
200 – SAG Milling	$30,219,700
240 – Ball Milling	$46,484,200
300 – Rougher/Scavenger Flotation	$11,677,500
315 – Cleaner Flotation	$10,179,600
350 – Concentrate Thickener	$12,342,700
400 – Gold Room & EW	$5,353,900
500 – Tailings Thickener	$13,848,300
800 - Reagents	$9,105,100
900 – Services (Air, Water)	$11,826,600
960 – Transportation to Site	$10,550,700
970 - Buildings	$11,971,700
980 – Mobile Equipment Allowance	$745,000
Subtotal Process Plant Direct Costs	**$212,543,800**

It should be noted that in the distribution of capital, 10% of the Process Direct Capital cost falls into Year 1. This is the hold back on the commissioning of the plant to commercial production levels.

The Indirect capital costs are provided in Table 21-12.

Table 21-12: Process Indirect Capital Costs ($CDN)

Area	Total Cost, $CDN
Preproduction & Site Costs	$8,716,700
First Fills	$5,386,100
EPCM & Contracts	$26,045,300
Mechanical & Electrical Spares	$4,310,000
Sustaining Capital	$1,419,600
Subtotal Process Plant Indirect Costs	**$45,877,700**

Process contingencies have been applied to the various areas and total $48.9 million.

The total process capital cost is outlined in Table 21-13.

Table 21-13: Process Capital Cost ($CDN)

Cost Category	Total Cost, $CDN
Direct Capital	$212,543,800
Indirect Capital	$45,877,700
Contingency	$48,923,700
Grand Total	**$307,345,300**

Estimating Methodology

The capital cost estimate was based on process designs and equipment lists plus preliminary plant layouts. The preliminary plant layout enabled an area by area assessment of quantities and factors for earthworks, concrete, steelwork, platework and electrical disciplines. Unit rates were established for bulk materials, capital equipment and labour from current in-house data and discussions with contractors.

21.2.4 Process Direct Costs

Mechanical Equipment

The costs for major mechanical equipment items such as crushers, mills (primary and regrind), thickeners and flotation cells were determined through discussions with equipment vendors. Smaller equipment items such as pumps, agitators, cyclones, reagent dosing systems etc. were estimated using older budget quotes (escalated to 2020) and AGP database costs. Installation costs for items of equipment were developed using estimates of man-hours and labour/equipment rates from similar (Northern climate) recent projects. Transportation and offloading costs for each piece of equipment are estimated using volumetric mass estimates together with per load rates to site.

Civil and Earthworks

Estimates of civil and earthwork quantities were made in each plant area using layout sketches in each process area, together with compound supply and installation rates derived using detailed information from similar projects.

Structural Steel and Platework

An estimate of platework and structural steel costs was made by calculating the approximate mass of major steel items, using rough dimensions together with detailed information from recent projects of similar size and scope. The direct cost includes transportation/delivery to site. Structural steel erection costs were derived using compound rates and calculated masses.

Piping

Piping costs were factored for each plant area using data derived from numerous other mineral processing projects and studies. The direct cost includes transportation to site and installation.

Electrical and Instrumentation

Electrical and Instrumentation costs were factored for each plant area using data derived from numerous other mineral processing projects and studies. The direct cost includes transportation to site and installation.

Building Costs

Building costs were built up using area/volume estimates for this project and by using detailed cost information from AGP's database of other similar projects. Building costs include insulation and HVAC. Direct costs include the transportation of materials to site and installation using local contractors.

21.2.5 Process Indirect Costs

Contractor Indirect Costs

Contractor indirect costs encompass the remaining cost of installation. It is based on a percentage of direct plant project costs and for this project is estimated to be approximately 4% of the direct costs. It typically includes items such as offsite management, onsite staff and supervision above trade level, crane drivers, specialized construction equipment and general labour mobilization and demobilization.

Pre-production/Commissioning Costs

Commissioning costs have been estimated based on typical unit rates for vendor representative labour as well as consideration for preproduction operations labour to allow for sufficient training and orientation of operations personnel prior to commencing production. The following rationale was applied:

- preproduction operations labour assumed at 8 weeks for operations personnel
- up to 20 vendors on site for 10 days each, including a travel allowance for each vendor visit

First Fills

A provision for the first fill of consumables (grinding media, lubricants, fuels, and reagents) is included in the estimate based on assumptions of quantities required and their respective unit costs.

Spare Parts

A provision for the initial stocking of critical mechanical and electrical spare parts has been made, estimated to be approximately 6% of the mechanical equipment costs.

Engineering Services

A lump sum allowance for engineering, procurement, construction management (EPCM) services was included, calculated using a factor of 12% of plant direct area costs (based on project location, complexity, and size). An additional allowance was made for external services and contracts.

Exclusions

The following items are specifically excluded from the process plant capital cost estimate:

- permits and licences
- project sunk costs
- government and local taxes and duties
- exchange rate variations
- mining and tailings capital costs
- owners/head office costs

21.2.6 Infrastructure Capital Costs

While the Troilus project is the beneficiary of significant in place infrastructure from previous mining, there are still various items that need to be built as part of the operation. These have been outlined in Table 21-14.

Table 21-14: Troilus Project Capital ($CDN)

Equipment	Preproduction Year -2, -1 ($CDN)	Sustaining LOM ($CDN)	Total ($CDN)
Pit Surface Power Distribution	150,000	-	150,000
Mine Dry	1,600,000	-	1,600,000
Mine Maintenance Shop (10 bay)/Warehouse	12,000,000	3,000,000	15,000,000
Shop Supplies/Tools/Equipment, Wash bay equipment	1,800,000	-	1,800,000
Explosives Plant Pad, Services and Storage	475,000	-	475,000
Fuel Storage	515,000	-	515,000
Reagent and Cold storage	350,000	-	
Assay Laboratory (Mine/Plant)	2,500,000	-	2,500,000
Permanent Camp	4,500,000	-	4,500,000
Administration Building	1,500,000	-	1,500,000
Gate Houses	70,000	-	70,000
Surface Water Management Ditches and ponds	750,000	-	750,000
Spill Containment Pond	200,000	-	200,000
Landfill Area	50,000	-	50,000
Truck Weigh Scale	240,000	-	240,000
Mobile Site Equipment	2,500,000	-	2,500,000
Tailings – Initial Pond Preparation	750,000	-	750,000
Tailings – Annual Lift Expenses	-	19,248,000	19,248,000
Tailings – Reclaim Water System	1,200,000	-	1,200,000
Tailings – Pipelines	1,100,000	-	1,100,000
Tailings – Water Treatment Plant	500,000	-	500,000
No Name Diversion Ditch	3,917,000	-	3,917,000
Dump Facility Preparation	500,000	-	500,000
Dump Diversion Ditch	75,000	-	75,000
Settling Ponds – below waste dumps	500,000	-	500,000
Mine Water Treatment Plant	2,000,000	-	2,000,000
Main Access Road to Site – Relocation of 5.2 kilometers	520,000	-	520,000
Pit Access Roads – 2 kilometers	400,000	-	400,000
Communications System	200,000	-	200,000
No Name Diversion Dyke – Initial Contractor Portion	500,000	-	500,000
Total Infrastructure Capital	**42,062,000**	**22,248,000**	**64,310,000**

There are various items outlined in the Infrastructure capital. The major items are explained below:

- Mine Maintenance Shop – 10 bay facility with wash bay, tire bay and welding bay
- Explosives Plant Pad and Services – reactivation of previous explosives plant area
- Fuel Storage – sufficient for three days of fuel storage
- Assay Laboratory – sample preparation and analysis facility for RC and plant samples
- Permanent Camp
  - Vendor down payment per quotation for 300 person camp
  - Additional cost for the camp is covered in the cost per person rate
- Administration Building – offices and clinic for the various departments
- Mobile Equipment Site – smaller pieces of equipment for general site work
- Tailings Preparation
  - Initial – cost to reactivate the facility
  - Sustaining – annual cost for lifts tied into the mine placed buttress material
- Diversion Ditch
  - cost to construct 8.3 kilometres of ditch around mining area
  - Ditch is 4 metres deep with a base of 3 metres and top width of 19 metres
  - Cost includes access road alongside, crossover bridges for light vehicles and flow monitoring stations
- Dump Facility development
  - Includes reactivation of 87 dump,
  - Diversion ditches for control of drainage
  - Settling pond construction for runoff
- Water Treatment Plant - for Mine Dewatering
- Relocation of Mine Access Road
  - Road will be diverted south and east of SW Zone to avoid active mining area

21.2.7 Environmental Capital Costs

Concurrent reclamation of mine waste dumps is included in the mine operating cost. The backfill of the 87 pit with waste from the J Zone pits assists in the reclamation of that pit.

A cost allowance of $25 million has been included at the end of the open pit and underground mine to remove all facilities and dispose of correctly.

21.2.8 Indirect Capital Costs

The Indirect and Contingency values for the plant have already been discussed in Section 21.2.3. For the tabulation of costs, the Indirect Costs have been summarized into one category (Indirects) and are further shown in Table 21-15 as dollars and percentages.

Table 21-15: Troilus Project Indirect Costs and Percentages ($CDN)

Area	Initial ($CDN)	Sustaining LOM ($CDN)	Total ($CDN)	Indirect Percentage (%)
Open Pit Mining	1,880,000	233,000	2,113,000	1%
Underground Mining	-	-	-	0%
Processing	41,281,000	4,587,000	45,868,000	22%
Infrastructure	6,270,000	3,377,000	9,647,000	15%
Environmental	-	-	-	0%
Owners Cost	15,000,000	-	15,000,000	-
Total Indirects	**64,431,000**	**8,197,000**	**72,627,000**	

The costs associated with Indirects include:

- Contractor Indirect Costs
  - offsite management, various labour groups, PPE, travel, etc.
- Construction Support Temporary Services Allowance
  - temporary offices and warehouses
  - setups of water, sewer, and power systems
- Mobile Equipment and Heavy Lift Cranes
- Vendor Representatives
- First Fills
- Spare Parts
- Freight
- Engineering Services

Included within the Troilus Indirect cost is an estimate of the Owners cost. This has been estimated to be $15 million in the pre-production period. This would cover:

- Permits and Licences
- Project Team and Expenses
- Preproduction Labour
- Head Office Fees, Expenses and Consultants

21.2.9 Contingency Capital Costs

Varying amounts of contingency have been applied depending on the project area and level of estimation. This is to cover anticipated variances between the specific items allowed in the estimate and the final total installed project cost. The contingency does not cover scope changes, etc., or the listed qualifications and exclusions.

Contingency has been applied to the estimate as a percentage allowance and the percentages applied are shown in Table 21-16.

Table 21-16: Project Indirect and Contingency Percentages ($CDN)

Area	Initial ($CDN)	Sustaining LOM ($CDN)	Total ($CDN)	Contingency Percentage (%)
Open Pit Mining	375,000	47,000	422,000	5%
Underground Mining	-	25,579,000	25,579,000	12%
Processing	44,022,000	4,892,000	48,914,000	23%
Infrastructure	4,206,000	2,225,000	6,431,000	10%
Environmental	-	2,500,000	2,500,000	10%
Total Contingency	**48,604,000**	**35,242,000**	**83,846,000**	

21.3 Operating Cost Estimates

21.3.1 Operating Cost Summary

The estimated project operating costs are shown in Table 21-17 highlighting different periods. With the open pit complete in Year 14 the costs after reflecting only the underground mine in operation. All costs are Canadian Dollars (CDN) unless otherwise noted.

Table 21-17: Troilus Project Operating Costs ($CDN)

Area	Units	Open Pit Only (Year 1 – 5)	Open Pit and Underground (Year 1 – 14)	Underground Only (Year 15 – 22)	Life of Mine (Year 1-22)
Open Pit Mining	$CDN/t moved	2.73	2.70	-	2.70
	$CDN/t mill feed	15.62	12.62	-	12.62
Underground Mining	$CDN/t mill feed	-	19.54	19.26	19.38
Processing	$CDN/t mill feed	6.74	6.74	6.74	6.74
G&A	$CDN/t mill feed	1.48	1.60	4.19	1.92
Concentrate Trucking	$CDN/t mill feed	0.26	0.32	0.26	0.32
Total Operating Cost	**$CDN/t mill feed**	**24.10**	**22.05**	**30.45**	**23.08**

21.3.2 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 $1.03 per litre 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 Quebec. 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 40% was applied to all rates. Mine positions and salaries are shown in Table 21-18.

Table 21-18: Open Pit Mine Staffing Requirements and Annual Salaries (Year 5)

Staff Position	Employees	Full Load Annual Salary ($CDN/yr)
Mine Maintenance		
Maintenance Superintendent	1	210,000
Maintenance General Foreman	1	182,000
Maintenance Shift Foremen	4	147,000
Maintenance Planner/Contract Admin	2	133,000
Clerk/Secretary	1	84,000
Subtotal	**9**	
Mine Operations		
Mine Ops/Technical Superintendent	1	224,000
Mine Operations General Foreman	1	196,000
Mine Shift Foreman	4	147,000
Junior Shift Foreman	4	133,000
Road Crew/Services Foreman	1	147,000
Clerk/Secretary	1	84,000
Subtotal	**12**	
Mine Engineering		
Chief Engineer	1	196,000
Senior Engineer	1	168,000
Open Pit Planning Engineer	2	147,000
Geotech Engineer	1	147,000
Blasting Engineer	1	147,000
Blasting/Geotech Technician	2	98,000
Surveyor/Mining Technician	2	98,000
Surveyor/Mine Technician Helper	2	91,000
Clerk/Secretary	1	84,000
Subtotal	**14**	
Geology		
Chief Geologist	1	182,000
Senior Geologist	1	154,000
Grade Control Geologist/Modeler	2	126,000
Sampling/Geology Technician	4	98,000
Subtotal	**9**	
Total Mine Staff	**44**	

The mine staff labour remains consistent for the mine life after the initial recruitment in the pre-production period (Year -1). This level plateaus at 44 staff in Year 1, including Mine Operations, Maintenance, Engineering, and Geology. As the open pit mine is completed, the staff is reduced; this begins in Year 10 bringing the staff to a level of 40 people. In the final year, the staff contingent is only 25.

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 21-19.

Table 21-19: Hourly Manpower Requirements and Annual Salary (Year 5)

Hourly Position	Employees	Full Load Annual Salary ($CDN/yr)
Mine General		
General Equipment Operator	8	112,900
Road/Pump Crew	8	105,200
General Mine Laborer	8	98,600
Trainee	4	78,900
Light Duty Mechanic	3	150,900
Tire Man	4	115,000
Lube Truck Driver	4	115,000
Subtotal	**39**	
Mine Operations		
Driller	36	112,900
Blaster	2	112,900
Blaster’s Helper	4	101,900
Loader Operator	8	119,000
Hydraulic Shovel Operator	16	119,000
Haul Truck Driver	104	102,000
Dozer Operator	16	112,900
Grader Operator	6	112,900
Transfer Loader	3	112,900
Snowplow/Water Truck	8	102,000
Subtotal	**203**	
Mine Maintenance		
Heavy Duty Mechanic	49	143,400
Welder	30	143,400
Electrician	2	143,400
Apprentice	8	108,900
Subtotal	**89**	
Total Hourly	**331**	

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 clerk/secretary.

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 a clerk/secretary 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 clerk/secretary.

The Mine Maintenance Superintendent has the Maintenance General Foreman reporting to him. Four Mine Maintenance Shift Foremen will report to the Maintenance General Foreman. As well, there are two maintenance planners/contract administrators and a clerk.

The hourly labour force includes positions for the light duty mechanic, tire men, and lube truck drivers. 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).

The drilling labour force is based on one operator per drill, per crew while operating. This on average is 16 drillers and holds there until Year 10 when it drops down over time as the drilling hours are diminished.

Shovel and loader operators peak at 24 in Year 1 and hold at that level until Year 10 when it also will start to tail off. Haulage truck drivers peak at 108 in Year 2 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 21-20.

Table 21-20: 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 5,000 hours per tire with proper rotation from front to back. On the haulage trucks each tire costs $29,500 so the cost per hour for tires is $35.40/hr 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 both mill feed and waste is estimated at 25 m/hr. Equipment costs used in the estimate are shown in Table 21-21.

Table 21-21: Major Equipment Operating Costs – no labour ($CDN/hr)

Equipment	Fuel/ Power	Lube/ Oil	Tires	Under-Carriage	Repair & Maint.	GET/ Consumables	Total
Production Drill	100.70	10.07	-	6.00	66.20	69.65	252.62
Production Loader	116.60	11.66	67.20	-	121.13	15.00	331.59
Hydraulic Shovel	35.18	-	-	-	180.81	13.00	228.99
Haulage Truck – 181 t	114.48	11.45	35.40	-	67.24	5.00	233.57
Crusher Loader	79.50	7.95	23.20	-	73.27	10.00	193.92
Track Dozer	74.20	7.42	-	10.00	64.04	5.00	160.66
Grader	23.32	2.33	4.00	-	18.58	5.00	53.23

Drilling in the open pit will be performed using conventional down the hole (DTH) blasthole rigs with 200 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 21-22.

Table 21-22: Drill Pattern Specification

Specification	Unit	Mill Feed	Waste
Bench Height	M	10	10
Sub-Drill	M	1.1	1.1
Blasthole Diameter	mm	200	200
Pattern Spacing – Staggered	m	6.2	6.2
Pattern Burden – Staggered	m	5.4	5.4
Hole Depth	m	11.1	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 21-23 are the parameters used for estimating drill productivity. The drill is configured for single pass drilling of the 11.1 metre deep hole.

Table 21-23: Drill Productivity Criteria

Drill Activity	Unit	Mill Feed	Waste
Pure Penetration Rate	m/min	0.50	0.50
Hole Depth	m	11.1	11.1
Drill Time	min	22.20	22.20
Move, Spot, and Collar Blasthole	min	3.00	3.00
Level Drill	min	0.50	0.50
Pull Drill Rods	min	1.00	1.00
Total Setup/Breakdown Time	min	4.50	4.50
Total Drill Time per Hole	min	26.7	26.7
Drill Productivity	m/h	24.9	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 21-24.

Table 21-24: Design Powder Factors

	Unit	Mill Feed	Waste
Powder Factor	kg/m3	0.859	0.859
Powder Factor	kg/t	0.30	0.30

The blasting cost is estimated using quotations from a local vendor. The emulsion price is $84.76/100 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 $202,000/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 21-25. This highlights the focus on the shovels over the loaders.

Table 21-25: Loading Parameters – Year 5

	Unit	Front-End Loader	Hydraulic Shovel
Bucket Capacity	m3	23.0	22.0
Waste Tonnage Loaded	%	18	82
Mill Feed Tonnage Mined	%	30	70
Bucket Fill Factor	%	95	95
Cycle Time	Sec	40	38
Trucks Present at the Loading Unit	%	80	80
Loading Time	min	2.8	3.23

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 21-26.

Table 21-26: 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	16	12.1	30	30	20
Empty (km/h)	50	40	35	25	35	35	20

Support equipment hours and costs are determined using the percentages shown in Table 21-27.

Table 21-27: Support Equipment Operating Factors

Mine Equipment	Factor	Factor Units
Track Dozer	25%	Of haulage hours to a maximum of 6 dozers
Grader	10%	Of haulage hours to a maximum of 3 graders
Crusher Loader	40%	Of loading hours to maximum of 1 loader
Support Backhoe	10%	Of loading hours to maximum of 2 backhoes
Water Truck	10%	Of haulage hours to a maximum of 2 trucks
Tire Manipulator	2	hours/day
Lube/Fuel Truck	6	hours/day
Mechanic's Truck	14	hours/day
Welding Truck	8	hours/day
Blasting Loader	8	hours/day
Blaster's Truck	8	hours/day
Integrated Tool Carrier	4	hours/day
Compactor	2	hours/day
Lighting Plants	12	hours/day
Pickup Trucks	10	hours/day
Dump Truck – 20 ton	6	hours/day

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 10m x 5m 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 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 1.1 million metres of drilling are expected to be completed for grade control work.

A total of 1.3 million samples will be assayed from that drilling at a cost of $15/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 $5 million per year with a peak of $CDN 7.5 million in Year 6.

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. Both Caterpillar and Komatsu have the ability, and desire, to allow financing of their product lines.

Indicative terms for leasing provided by the vendors are:

- Down payment = 0% of equipment cost – if the entire fleet selection was theirs
- Term Length = variable between 2 and 5 years depending on equipment type
- Interest Rate = LIBOR plus a percentage
- Residual = $0

The proposed interest rate is used to calculate a multiplier on the amount being leased. The multiplier is 1.067 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 no down payment was required, the full finance 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 $CDN 0.41/t to the mine operating cost over the life of the mine. On a cost per tonne of mill feed basis, it was $CDN 1.91/t mill feed.

Dewatering

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 Troilus 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 12,000 m^3/day on average will need to be pumped from within the pit. 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 15,000 m^3/day 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 are 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 $1.8 M over the life of the mine.

The dewatering operating cost is estimated at $3.1 million over the mine life or $272,000 per year.

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 21-28 and Table 21-29.

Table 21-28: Open Pit Mine Operating Costs – with Finance Cost ($CDN/t Total Material)

Open Pit Operating Category	Unit	Year 1	Year 3	Year 5	LOM Average Cost
General Mine and Engineering	$CDN/t	0.18	0.20	0.20	0.23
Drilling	$CDN/t	0.24	0.25	0.26	0.24
Blasting	$CDN/t	0.47	0.48	0.51	0.50
Loading	$CDN/t	0.17	0.18	0.17	0.18
Hauling	$CDN/t	0.63	0.76	0.73	0.75
Support	$CDN/t	0.21	0.22	0.22	0.27
Grade Control	$CDN/t	0.07	0.10	0.10	0.10
Finance Costs	$CDN/t	0.65	0.74	0.37	0.41
Dewatering	$CDN/t	0.00	0.01	0.01	0.01
Total	**$CDN/t**	**2.63**	**2.92**	**2.58**	**2.70**

Table 21-29: Open Pit Mine Operating Costs – with Finance Cost ($CDN/t Mill Feed)

Open Pit Operating Category	Unit	Year 1	Year 3	Year 5	LOM Average Cost
General Mine and Engineering	$CDN/t mill feed	1.13	1.10	1.12	1.09
Drilling	$CDN/t mill feed	1.50	1.37	1.45	1.14
Blasting	$CDN/t mill feed	2.90	2.69	2.84	2.33
Loading	$CDN/t mill feed	1.05	0.97	0.97	0.84
Hauling	$CDN/t mill feed	3.91	421	4.08	3.52
Support	$CDN/t mill feed	1.28	1.20	1.20	1.28
Grade Control	$CDN/t mill feed	0.41	0.54	0.58	0.47
Finance Costs	$CDN/t mill feed	4.04	4.10	2.06	1.91
Dewatering	$CDN/t mill feed	0.02	0.03	0.03	0.04
Total	**$CDN/t mill feed**	**16.25**	**16.21**	**14.33**	**12.62**

Underground Mine Operating Cost Estimate

The operating period commences at the start of 3rd quarter Year 9 at the end of the project capital period. A summary of the mine operating cost estimate is provided by element and by activity in Table 21-30. Life of mine unit operating costs by element and by activity are shown graphically in Figure 21-3 and Figure 21-4 respectively.

Table 21-30: Summary of Life of Underground Mine Operating Costs

	$CDN 000's	$CDN/t
OPERATING COSTS BY ELEMENT		
Labour	387,023	9.15
Supplies	201,503	4.76
Equipment	163,923	3.87
Fuel	43,866	1.04
Power	23,774	0.56
Contingency	-	-
Total	**820,089**	**19.38**
OPERATING COSTS BY ACTIVITY		
Operating Development in Waste	15,453	0.37
Ore Development	144,915	3.42
Stoping & Mucking	180,935	4.28
Truck Haulage	11,661	0.28
Delineation Drilling	11,030	0.26
Grizzley & Rail Veyor	108,619	2.57
Mobile Equipment Leasing	55,248	1.31
Mine Services	148,938	3.52
Supervision & Technical	101,151	2.39
Mine Air Heating	18,364	0.43
Power	23,774	0.56
Contingency	-	-
Total	**820,089**	**19.38**

Figure 21-3: Life of Mine Unit Operating Cost by Element

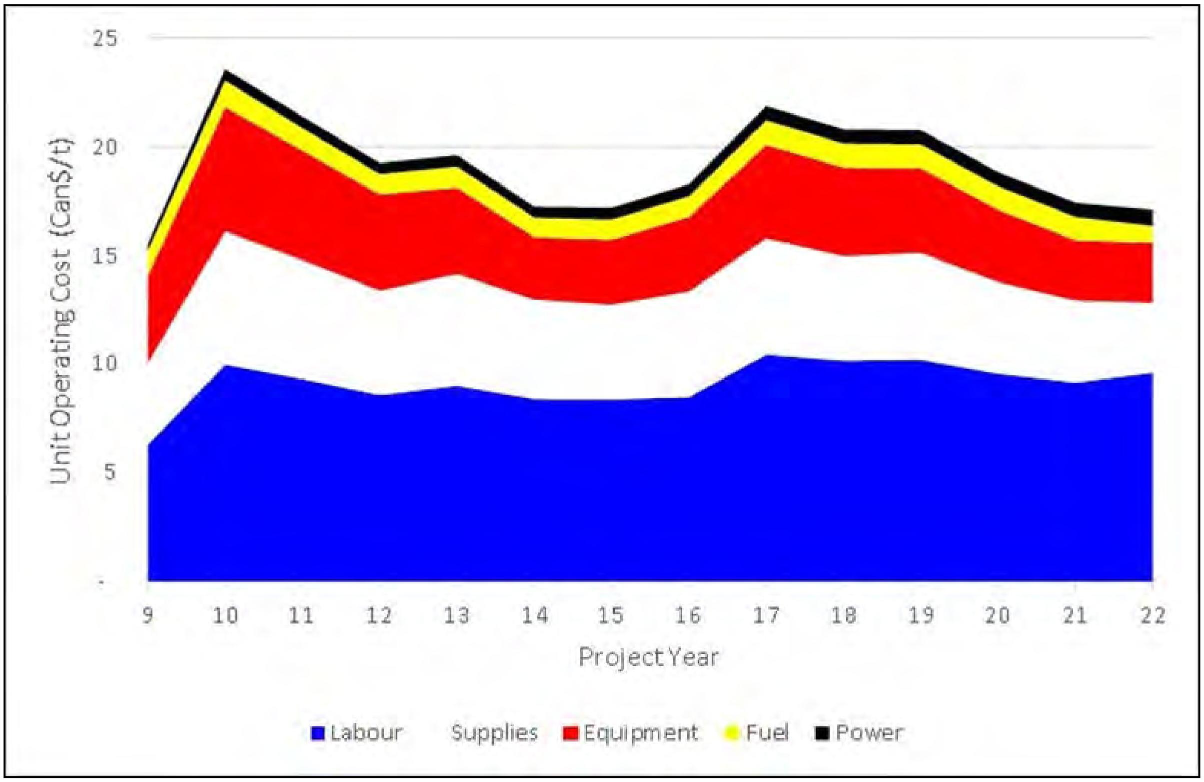
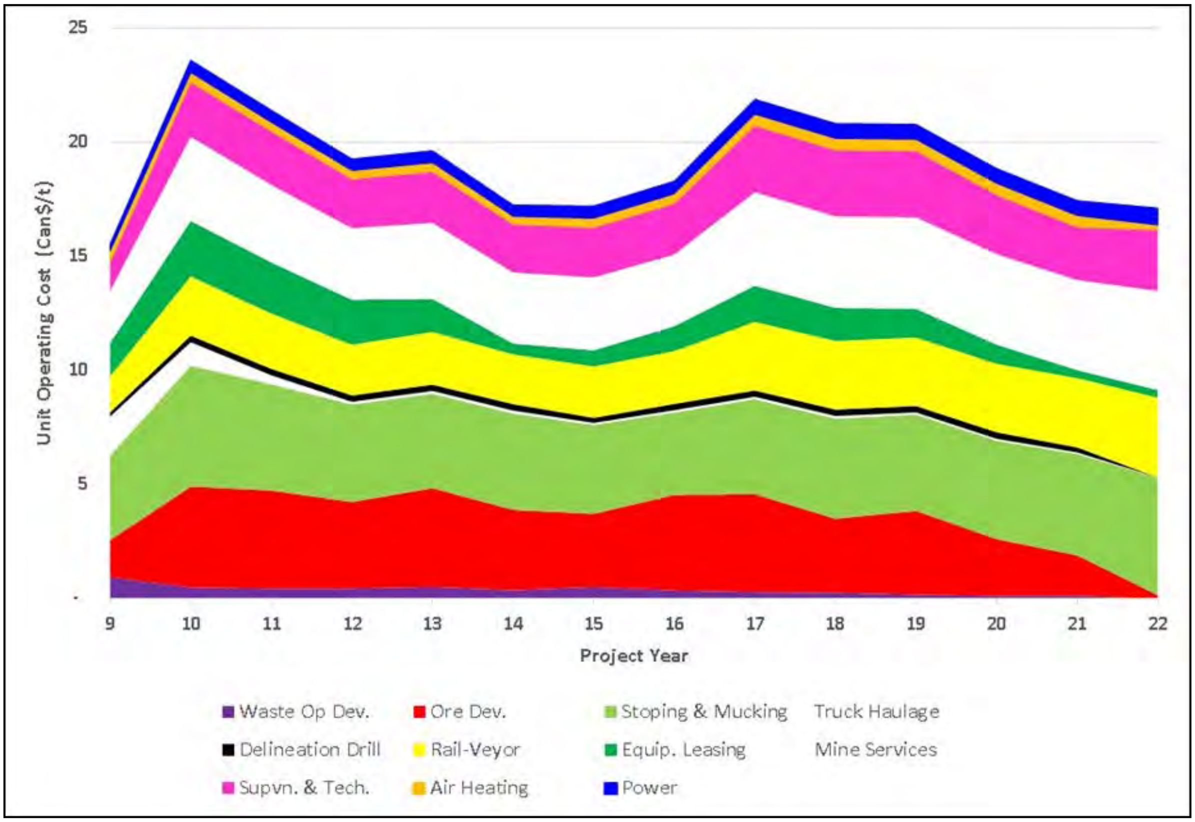


Figure 21-4: Base Case Life of Mine Unit Operating Cost by Activity



21.3.3 Process Operating Costs

The process plant operating costs have been developed based on a design processing rate of 12.6 Mtpa of material. The plant will normally operate 24 hrs/day-365 days/year with 6,570 hrs/year for the primary crusher and 8,077 hrs/year for grinding flotation circuits. All costs are expressed in Canadian dollars unless otherwise stated, to an accuracy of ±35% and are based on Q3 2020 pricing. The process plant operating costs are broken down into fixed and variable costs and are summarized below in Table 21-31. No contingency has been added to the operating cost.

Table 21-31: Process Operating Costs ($CDN)

Fixed Costs	Annual Cost, $CDN	
	$CDN 000's	per d.m.t
Labour (plant and laboratory)	$9,965	$0.78
Tools/equipment/safety	$525	$0.04
Plant maintenance and spares	$5,720	$0.45
Contracts	$450	$0.04
Fuel	$435	$0.03
Assay and general laboratory	$683	$0.05
Fixed Cost Subtotal	**$17,778**	**$1.39**
Variable Costs	**Annual Cost, $CDN**	
	$CDN 000's	**per d.m.t**
Power	$11,180	$0.88
Reagents	$14,795	$1.16
Mill balls	$22,918	$1.79
Liners (crushers and mills)	$15,143	$1.19
Process plant piping	$1,022	$0.08
Contracts	$1,200	$0.09
Lubricants	$330	$0.03
Abnormal items	$1,789	$0.14
Variable Cost Subtotal	**$68,377**	**$5.35**
Total Operating Cost	**$86,155**	**$6.74**

The process operating costs were developed in accordance with industry practice for PEA of copper and gold processing plants. Quantities and cost data were compiled from a variety of sources including:

- historical metallurgical testwork
- consumable prices from suppliers
- AGP internal data
- first principal calculations

The following major categories were used to estimate the plant 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 mill liners, grinding media, cyclone parts, screen panels crusher and mill lubricants, and tailings 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 mill liners and grinding media and were predicted based on the material bond abrasion index and the mill power consumption.
- Reagent consumptions were derived from historical metallurgical testwork, and adjusted, where experience deemed necessary to reflect actual plant operating practice.
- Fuel consumption for mobile equipment is based on standard fuel consumption rates and equipment utilization.
- Reagent prices were derived from reagent supplier quotes or AGP database of recent projects.

Maintenance

Maintenance costs exclusive of labour and consumable costs were estimated as a factor of mechanical supply costs. A factor of 6.5% was used and is consistent with AGP 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 44 MW and process plant will be operated for 7,700 hours per annum for a total annual power consumption of 338.8 MWh. The cost of power was estimated at $CDN 33/MWh.

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 the AGP project database. The process plant labour includes a combination of day and shift work. A summary of the labour complement is provided below in Table 21-32.

Table 21-32: Process Labour

Department	No. of Employees
Management	2
Admin	4
Metallurgy	2
Laboratory/Sample Preparation	5
Operations	14
Maintenance	26
Total	**53**

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 are associated with analysis of routine plant samples to monitor plant 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 $0.05/t material processed.

Exclusions

The process plant operating cost estimate excludes the following items:

- ROM and material handling costs
- government monitoring/compliance costs
- head office and corporate overhead costs
- local and federal government taxes and duties
- impacts of foreign exchange rates
- licence and union fees
- insurance costs
- contingency

21.3.4 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 8 at $CDN 21.3 million per year.

A catering and accommodation quotation of $CDN 79 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 $CDN 11.9 million per year in Years 8 and 9. From that point in time the camp cost declined the open pit feed rates dropped due to greater production from the underground mine.

Wages for staff and hourly personnel in the G&A area totaled $CDN 5.8 million per year.

Bussing from Chibougamau and other areas of personnel peaked at $CDN 1.7 million per year in Year 8.

The life of mine average G&A cost was $CDN 1.92 per tonne of mill feed or $CDN 369.4 million total.

21.3.5 Concentrate Trucking

The cost of trucking was determined by contacting various rail and trucking companies. From these inquiries, an estimate of $CDN 70.30 per wet metric tonne of concentrate was used in the calculations with the assumption of trucking. Transportation from the Troilus site to the Horne Smelter was assumed.

22 ECONOMIC ANALYSIS

22.1 Introduction

A Preliminary Economic Assessment (PEA) of the Troilus Gold Project has been conducted using a simple pre-tax and post-tax cash flow model prepared by AGP on behalf of Troilus Gold. The model was created in Excel for Troilus' use.

The PEA was prepared in accordance with NI 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101"). Readers are cautioned that the PEA is preliminary in nature. It includes inferred mineral resources considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty the PEA will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, socio-political, marketing, or other relevant issues.

The quantity of Indicated and Inferred material reported in Section 16 and used in the economic analysis of the Project are shown in Table 22-1. Approximately 75% of the feed material currently is at the Indicated resource classification.

Table 22-1: Mill Feed Resource Classification

Mining Type	Area	Indicated					Inferred				
		Mill Feed (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	AuEq (g/t)	Mill Feed (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	AuEq (g/t)
Open Pit	Z87	34.7	0.73	0.090	1.40	0.86	1.9	0.55	0.063	1.39	0.64
	J	79.3	0.52	0.060	0.90	0.61	15.5	0.45	0.062	0.82	0.54
	SW	0.0	0.00	0.000	0.00	0.00	18.8	0.64	0.065	0.76	0.74
	Subtotal	**114.0**	**0.59**	**0.069**	**1.05**	**0.69**	**36.2**	**0.56**	**0.064**	**0.82**	**0.65**
Underground	Z87	30.1	1.17	0.111	1.08	1.34	12.2	1.21	0.093	0.33	1.36
Total	**Tonnes/grade**	**144.1**	**0.71**	**0.078**	**1.06**	**0.82**	**48.4**	**0.72**	**0.071**	**0.70**	**0.83**
	%	**74.9%**					**25.1%**				

All pricing is H2 2020 Canadian dollars unless otherwise noted.

The decision to use a production rate of 35,000 tpd was determined earlier in the study. This rate provided a reasonable net present value (NPV), mine life and initial capital below $500 million; criteria provided by Troilus Gold.

Two options were examined for development of the Troilus Gold Project:

1. Open Pit with Underground
2. Open Pit only – discussed further in Section 24

The decision to use the Open Pit with Underground option as the Base Case was made based on:

- higher net present value
- no sterilization of potential future resources due to larger waste storage facilities covering them
- reduced environmental footprint

The footprint results from the ability to partially backfill the 87 pit with waste from the J Zone pit phases. This results in smaller and lower waste storage facilities and fewer empty pits upon completion.

The results of the financial analysis using discounted cash flow is summarized in Table 22-2. All calculations and data was collected in Canadian dollars.

Table 22-2: Troilus Gold Project – Discounted Cash Flow Financial Summary ($CDN)

Parameter	Units	Pre-Tax	Post-Tax
Metal Prices			
Gold	$US/oz	1,475.00	
Copper	$US/lb	3.00	
Silver	$US/oz	20.00	
Exchange Rate	$CDN:$US	0.74	
Net Present Value (5%)	$CDN M	$1,311	**$778**
Internal Rate of Return	%	29.6	**22.9**
Net Revenue less Royalties	$CDN M	8,322.4	**8,322.4**
Total Operating Cost	$CDN M	4,443.0	**4,443.0**
Life of Mine Capital Cost	$CDN M	1,132.1	**1,132.1**
Taxes	$CDN M	-	**1,038.8**
Net Cash Flow	$CDN M	2,747.3	**1,708.5**
Payback Period	Years	3.7	**4.0**
Cash Costs (with credits)	$CDN/oz	970	**1,241**
All-in Sustaining Cost	$CDN/oz	1,148	**1,419**
Payable Metals (Life of Mine)			
Gold	Moz	3.84	
Copper	M Lbs	265	
Silver	Moz	1.47	
Initial Capital	$CDN M	449.5	
Sustaining Capital	$CDN M	682.6	
Total Capital	$CDN M	1,132.1	
Mine Life	Years	21	

For readers preferring to compare projects in United States dollars, Table 22-3 has been prepared using an exchange rate of 0.74 ($CDN:$US).

Table 22-3: Troilus Gold Project – Discounted Cash Flow Financial Summary ($USD)

Parameter	Units	Pre-Tax	Post-Tax
Metal Prices			
Gold	$US/oz	1,475.00	
Copper	$US/lb	3.00	
Silver	$US/oz	20.00	
Exchange Rate	$CDN:$US	0.74	
Net Present Value (5%)	$USD M	971	**576**
Internal Rate of Return	%	29.6	**22.9**
Net Revenue less Royalties	$USD M	6,165	**6,165**
Total Operating Cost	$USD M	3,291	**3,291**
Life of Mine Capital Cost	$USD M	839	**839**
Taxes	$USD M	-	**770**
Net Cash Flow	$USD M	2,035	**1,266**
Payback Period	Years	3.7	**4.0**
Cash Costs (with credits)	$USD/oz	719	**919**
All-in Sustaining Cost	$USD/oz	850	**1,051**
Payable Metals (Life of Mine)			
Gold	Moz	3.84	
Copper	M Lbs	265	
Silver	Moz	1.47	
Initial Capital	$USD M	333.0	
Sustaining Capital	$USD M	505.6	
Total Capital	$USD M	838.6	
Mine Life	Years	21	

22.2 Discounted Cash Flow Analysis

The Discounted Cash Flow (DCF) analysis was completed using the Base Case Parameters shown in Table 22-4.

Table 22-4: Discounted Cash Flow – Parameters

Parameter	Units	Value
Metal Prices		
Gold	$US/oz	1,475.00
Copper	$US/lb	3.00
Silver	$US/oz	20.00
Exchange Rate	$CDN:$US	0.74
Royalties – all metals	%	3.5
Net Present Value Discount Rate	%	5.0

22.2.1 Mineral Resource and Mine Life

The mineral resource used in the analysis are from both the open pit and the underground mines. They are discussed in Section 16. The tonnage and grades are shown again in Table 22-5.

Table 22-5: Mine Feed Tonnages and Grade

Item	Unit	Open Pit	Underground	Total
Mill Feed	Mt	150.2	42.3	192.5
Gold grade	g/t	0.58	1.18	0.71
Copper Grade	%	0.07	0.11	0.08
Silver Grade	g/t	1.00	0.86	0.97
Waste	Mt	591.1		
Life of Mine Strip Ratio	Waste: Mill Feed	3.9		

22.2.2 Metallurgical Recoveries and Concentrate Grades

Metallurgical recoveries used in the analysis are dependent on the area mined and the grade of the incoming feed. The potential copper concentrate grades are also dependent on the area. The weighted average of the feed origin is used to determine the recoveries and the appropriate concentrate grade.

The information for the project is summarized in Table 22-6.

Table 22-6: Metallurgical Recoveries and Concentrate Grades by Zone

Item	Unit	87 Zone	J Zone	SW Zone High Grade	SW Zone Low Grade
Concentrate Grade	%	23	12	19	17
				Cu > 0.13%	Cu < 0.13%
Copper Recovery	%	90	90	92	90
Gold Recovery				Au > 1.2 g/t	Au < 1.2 g/t
Gravity Recovery	%	30	30	30	30
Flotation Recovery	%	60	60	60	58
Total Gold Recovery	%	90	90	90	88
Silver Recovery	%	40	40	40	40

22.2.3 Smelting and Refining Terms

The smelting and refining terms used in the analysis reflect current market conditions and the expected range of copper concentrate grade from Troilus. The Life of Mine average is shown in Table 22-7 and the variable terms are shown in Table 22-8.

Table 22-7: Smelting and Refining Terms - LOM

Term	Unit	Copper	Gold	Silver
Cu, Au Minimum Deduction	%, g/dmt	1.2%	0.0 g/dmt	30.0 g/dmt
Base Smelting Charge	$US/dmt	62.00	-	
Cu Refining Charge	$US/lb payable	0.062	-	
Payable	%	100.0	96.5	90.0
Refining Charge	$US/oz	-	5.00	0.50
Concentrate Grade (LOM average)	%, g/t	16.6%	103 gpt	94 gpt
Concentrate Moisture	%	8.0	-	

Table 22-8: Copper Concentrate Variable Terms

Concentrate Grade (Cu%)	Copper Payable (%)	Copper Deduction (%Cu units)
12% - 14%	100%	1.4 units
14% - 16%	100%	1.3 units
16% - 18%	100%	1.2 units
18% - 20%	100%	1.1 units
20% - 22%	96.5%	1.0 units

For the gold dore produced from the gravity concentrator, the following terms were applied:

- Gold Payable = 99%
- Dore Selling Cost = $12 US/oz

22.2.4 Operating Costs

The mining, processing, administration and concentrate haulage 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 ($CDN)

Cost Area	Cost ($CDN M)	Unit Cost ($CDN/t Mill Feed)
Open Pit Mining	1,895.5	12.62
Underground Mining	820.1	19.38
Total Mining Cost	2,715.6	14.11
Processing	1,297.3	6.74
General and Administration	369.4	1.92
Concentrate Trucking	60.7	0.32
Total Operating Cost	**4,443.0**	**23.08**

22.2.5 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 ($CDN)

Area	Initial Capital (M$CDN)	Sustaining Capital (M$CDN)	Total Capital (M$CDN)
Open Pit – Prestrip (capitalized)	94.7	-	94.7
Open Pit - Capital	8.4	6.7	15.2
Open Pit Mining - Subtotal	103.1	6.7	109.9
Underground Mining	-	559.7	559.7
Processing	191.3	25.5	216.8
Infrastructure	42.1	22.2	64.3
Environmental	-	25.0	25.0
Indirects	64.4	8.2	72.6
Contingency	48.6	35.2	83.8
Total	**449.5**	**682.6**	**1,132.1**

22.2.6 Royalties

Royalties on all the metals is set at 3.5% per current Troilus Gold agreements.

22.2.7 Depreciation

All capital expenditures are depreciated depending on the class of asset.

22.2.8 Taxes

A taxation model was developed that considered the proper taxation for the jurisdiction of Quebec 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. Quebec Taxes = $CDN 706.7 million
2. Federal Taxes = $CDN 332.1 million
3. Total Taxes = $CDN 1,038.8 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.

22.2.9 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

22.3 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)

NPV $CDN M	-20%	-10%	Base Case	+10%	+10%
Metal Price	193	488	778	1,061	1,341
Capital Cost	902	840	778	715	653
Operating Cost	1,081	930	778	621	463
Exchange Rate	155	473	778	1,071	1,358

Table 22-12: IRR Sensitivity (Post-Tax)

IRR %	-20%	-10%	Base Case	+10%	+10%
Metal Price	9.2	16.5	22.9	28.8	34.4
Capital Cost	28.6	25.5	22.9	20.5	18.5
Operating Cost	30.1	26.5	22.9	19.1	15.4
Exchange Rate	8.6	16.0	22.9	29.1	35.0

The project is most sensitive to changes in metal price and exchange rate. The two are linked as the project is determined in Canadian dollars while the commodity price uses a world price in United States dollars.

The sensitivities are also show graphically in Figure 22-1 and Figure 22-2.

Figure 22-1: NPV Sensitivity (Post-Tax)

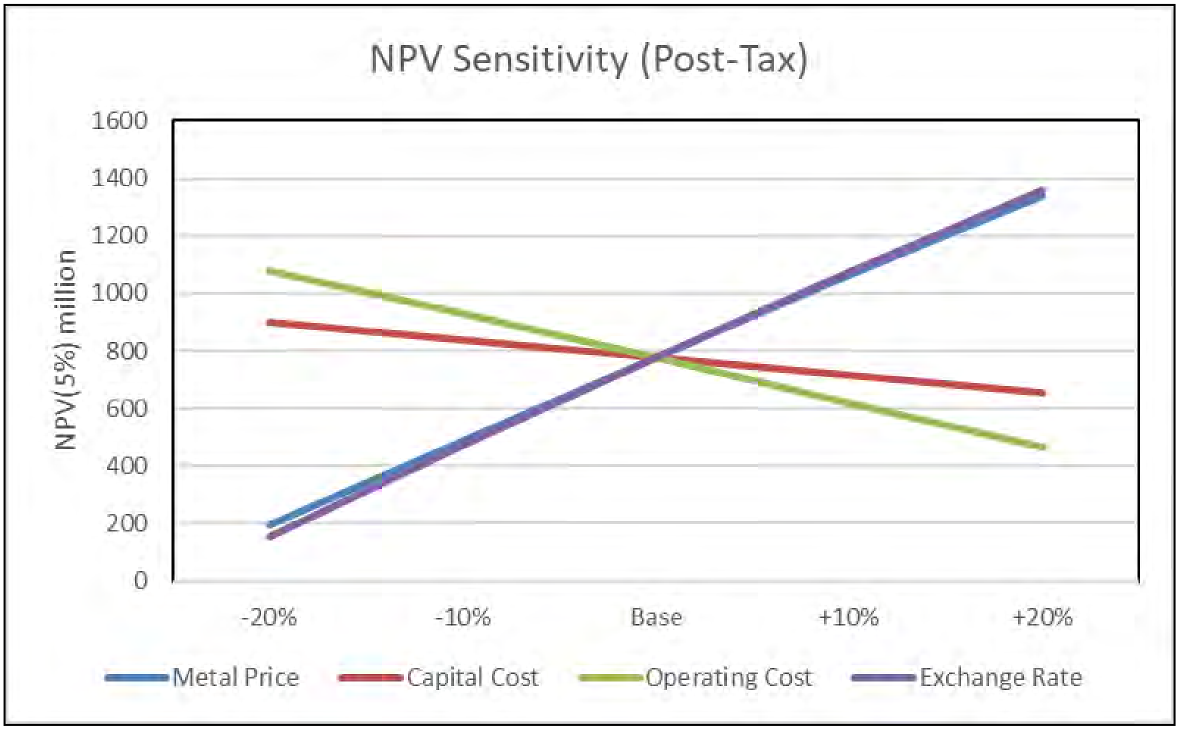
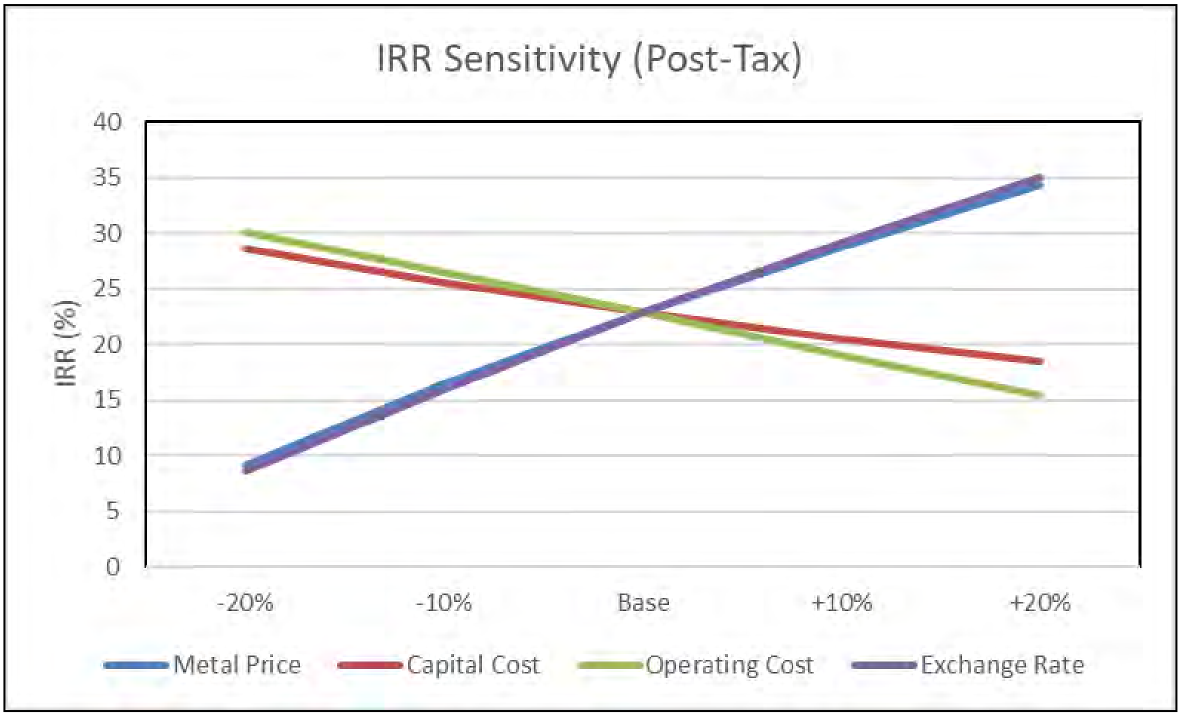


Figure 22-2: IRR Sensitivity (Post-Tax)



22.4 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 -2 to Year 10)

		Total		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	192,479,574		-	-	11,812,500	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000
Gold Equivalent	gpt	0.825		-	-	0.64	0.66	0.65	0.76	0.99	0.78	0.70	0.64	0.73	0.76
Gold	gpt	0.711		-	-	0.55	0.55	0.55	0.65	0.85	0.67	0.60	0.55	0.63	0.64
Copper	%	0.076		-	-	0.06	0.07	0.07	0.07	0.09	0.07	0.07	0.06	0.07	0.08
Silver	gpt	0.967		-	-	0.94	1.15	1.15	1.17	1.13	1.05	1.04	0.91	0.97	0.99
87 Pit Feed	tonnes	36,607,114				6,356,585	7,132,517	6,069,156	7,432,593	8,674,301	941,961	-	-	-	-
Percentage of feed	%	19.0%				53.8%	56.6%	48.2%	59.0%	68.8%	7.5%	0.0%	0.0%	0.0%	0.0%
J Pit	tonnes	94,780,095				-	2,268,397	6,530,533	4,126,264	1,133,218	9,731,095	11,440,930	11,861,771	10,022,868	8,792,866
Percentage of feed	%	49.2%				0.0%	18.0%	51.8%	32.7%	9.0%	77.2%	90.8%	94.1%	79.5%	69.8%
SW Pit	tonnes	18,776,538				5,455,915	3,199,085	311	1,041,143	2,792,482	1,926,944	1,159,070	632,378	374,553	652,565
Percentage of feed	%	9.8%				46.2%	25.4%	0.0%	8.3%	22.2%	15.3%	9.2%	5.0%	3.0%	5.2%
87 UG Feed	tonnes	42,315,827				-	-	-	-	-	-	-	105,851	2,202,578	3,154,569
Percentage of feed	%	22.0%				0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.8%	17.5%	25.0%
Recovered Metals															
Total															
Gold - Dore	ounces	1,320,728		-	-	62,746	67,069	66,831	79,030	103,490	81,879	73,032	66,495	76,130	77,459
Gold - Flotation	ounces	2,633,682		-	-	123,435	132,596	133,662	157,718	206,032	162,492	145,253	132,830	152,166	154,752
Gold Total	ounces	3,954,409		-	-	186,181	199,666	200,494	236,748	309,521	244,370	218,285	199,324	228,297	232,211
Copper	pounds	290,418,773		-	-	14,275,466	18,537,011	17,592,671	17,832,874	23,487,504	17,933,160	16,461,556	15,895,181	17,076,891	20,384,797
Silver	ounces	2,393,126		-	-	143,421	185,770	185,703	188,786	183,598	170,270	168,130	147,282	156,815	160,083
Cu Concentrate (DMT) - Grades	DMT	794,945				32,010	43,127	46,130	42,794	51,515	59,869	59,055	58,412	55,047	61,590
Gold	g/DMT	103.0				119.9	95.6	90.1	114.6	124.4	84.4	76.5	70.7	86.0	78.2
Copper	%	16.6%		0.0%	0.0%	20.2%	19.5%	17.3%	18.9%	20.7%	13.6%	12.6%	12.3%	14.1%	15.0%
Silver	g/DMT	93.6				139.4	134.0	125.2	137.2	110.9	88.5	88.6	78.4	88.6	80.8
Cu Concentrate (WMT)	WMT	864,070				34,794	46,878	50,142	46,515	55,994	65,075	64,190	63,491	59,834	66,945
Cu Concentrate Delivered to Smelter (less los	DMT	790,970				31,850	42,912	45,900	42,580	51,257	59,569	58,760	58,119	54,772	61,282
Cu Concentrate Payables															
Copper payables	%					96.5%	100.0%	100.0%	100.0%	96.5%	100.0%	100.0%	100.0%	100.0%	100.0%
Min Deduction	unit					1.0	1.1	1.2	1.1	1.0	1.4	1.4	1.4	1.3	1.3
Gold	ounces	2,528,795	96.0%			118,519	127,315	128,339	151,437	197,827	156,020	139,468	127,540	146,106	148,589
Copper	pounds	264,948,298	91.2%			13,029,348	17,403,679	16,290,409	16,711,112	21,461,636	16,004,907	14,565,655	14,021,866	15,421,743	18,526,540
Silver	ounces	1,474,083	61.6%			100,785	129,107	126,452	132,096	119,917	100,767	99,553	81,440	92,882	90,158
Dore Payables															
Gold	ounces	1,307,520	99.0%			62,119	66,399	66,163	78,240	102,455	81,060	72,301	65,830	75,369	76,684

Table 22-14: Detailed Cash Flow – Mill Production Calculations (Year 11 – 22)

		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
Mill Production														
Total Mill Feed	tonnes	192,479,574	12,600,000	12,600,000	12,600,000	5,754,253	3,285,000	3,285,000	3,285,000	3,285,000	3,285,000	3,285,001	3,246,700	756,121
Gold Equivalent	gpt	0.825	0.74	0.80	0.85	1.04	1.14	1.16	1.24	1.29	1.44	1.35	1.55	1.42
Gold	gpt	0.711	0.62	0.68	0.74	0.91	0.99	1.00	1.10	1.15	1.30	1.23	1.43	1.31
Copper	%	0.076	0.08	0.08	0.08	0.09	0.10	0.10	0.09	0.09	0.09	0.08	0.08	0.07
Silver	gpt	0.967	1.03	1.09	1.06	1.11	0.79	0.75	0.23	0.39	0.15	0.10	0.07	0.01
87 Pit Feed	tonnes	36,607,114	-	-	-	-	-	-	-	-	-	-	-	-
Percentage of feed	%	19.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
J Pit	tonnes	94,780,095	8,220,757	8,991,266	9,315,001	2,345,131	-	-	-	-	-	-	-	-
Percentage of feed	%	49.2%	65.2%	71.4%	73.9%	40.8%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
SW Pit	tonnes	18,776,538	1,094,243	323,727	-	124,122	-	-	-	-	-	-	-	-
Percentage of feed	%	9.8%	8.7%	2.6%	0.0%	2.2%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
87 UG Feed	tonnes	42,315,827	3,285,001	3,285,007	3,284,999	3,285,000	3,285,000	3,285,000	3,285,000	3,285,000	3,285,000	3,285,001	3,246,700	756,121
Percentage of feed	%	22.0%	26.1%	26.1%	26.1%	57.1%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%	100.0%
Recovered Metals														
Total														
Gold - Dore	ounces	1,320,728	75,314	82,418	89,427	50,415	31,424	31,812	34,824	36,524	41,129	39,090	44,629	9,562
Gold - Flotation	ounces	2,633,682	150,351	164,754	178,853	100,798	62,849	63,624	69,648	73,048	82,257	78,181	89,258	19,125
Gold Total	ounces	3,954,409	225,664	247,172	268,280	151,213	94,273	95,435	104,472	109,572	123,386	117,271	133,887	28,687
Copper	pounds	290,418,773	20,772,940	19,644,437	18,854,210	10,066,193	6,404,997	6,393,981	5,811,454	5,950,110	5,776,837	5,034,156	5,194,220	1,038,127
Silver	ounces	2,393,126	167,463	177,395	171,442	81,909	33,245	31,771	9,817	16,285	6,425	4,280	3,124	112
Cu Concentrate (DMT) - Grades	DMT	794,945	61,576	59,418	57,521	24,832	12,632	12,610	11,461	11,734	11,393	9,928	10,244	2,047
Gold	g/DMT	103.0	75.9	86.2	96.7	126.3	154.8	156.9	189.0	193.6	224.6	244.9	271.0	290.5
Copper	%	16.6%	15.3%	15.0%	14.9%	18.4%	23.0%	23.0%	23.0%	23.0%	23.0%	23.0%	23.0%	23.0%
Silver	g/DMT	93.6	84.6	92.9	92.7	102.6	81.9	78.4	26.6	43.2	17.5	13.4	9.5	1.7
Cu Concentrate (WMT)	WMT	864,070	66,931	64,585	62,523	26,991	13,730	13,706	12,458	12,755	12,383	10,791	11,135	2,225
Cu Concentrate Delivered to Smelter (less los	DMT	790,970	61,268	59,121	57,233	24,708	12,568	12,547	11,404	11,676	11,336	9,878	10,193	2,037
Cu Concentrate Payables														
Copper payables	%		100.0%	100.0%	100.0%	100.0%	96.5%	96.5%	96.5%	96.5%	96.5%	96.5%	96.5%	96.5%
Min Deduction	unit		1.3	1.3	1.3	1.1	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Gold	ounces	2,528,795	144,363	158,193	171,730	96,784	60,346	61,090	66,874	70,139	78,982	75,067	85,703	18,363
Copper	pounds	264,948,298	18,913,118	17,851,795	17,119,627	9,416,683	5,882,530	5,872,413	5,337,403	5,464,749	5,305,610	4,623,511	4,770,518	953,446
Silver	ounces	1,474,083	96,778	107,536	103,844	51,902	18,860	17,559	-	4,448	-	-	-	-
Dore Payables														
Gold	ounces	1,307,520	74,560	81,594	88,532	49,911	31,110	31,494	34,476	36,159	40,717	38,699	44,183	9,467

Table 22-15: Detailed Cash Flow – Cost and Revenue Calculations (Year -2 – Year 10)($CDN)

Mine Production				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		150,163,747													
Mill Feed															
Feed to Mill	tonnes	136,780,780		-	-	10,862,634	11,166,773	12,600,000	10,888,075	12,600,000	12,600,000	12,538,953	10,747,512	9,349,458	7,619,617
Feed to Stockpile	tonnes	13,382,967		-	3,121,559	-	3,522	969,938	1,060,740	3,504,711	4,322,191	16,241	92,198	-	-
Stockpile to Mill	tonnes	13,382,967		-	-	949,866	1,433,227	-	1,711,925	-	-	61,047	1,746,637	1,047,964	1,825,814
Waste	tonnes	591,085,219		-	36,393,368	62,137,366	61,142,589	56,430,062	58,051,186	53,895,289	48,077,809	37,444,805	45,590,925	37,613,551	35,646,219
Total Material	tonnes	754,631,933		-	39,514,927	73,949,866	73,746,111	70,000,000	71,711,925	70,000,000	65,000,000	50,061,047	58,177,271	48,010,973	45,091,650
Strip Ratio		3.94		-	-	5.26	4.85	4.48	4.61	4.28	3.82	2.97	3.65	3.62	3.77
Underground															
Feed to Mill	tonnes	42,315,827											105,851	2,202,578	3,154,569
Operating Cost			$/t feed												
Open Pit Mining	dollars	1,895,501,838	12.62	Prestripping capitalized		191,968,822	201,915,471	204,241,123	192,923,280	180,541,837	151,076,331	114,576,957	132,494,981	121,951,165	120,588,055
Underground Mining	dollars	820,089,318	19.38								-	-	-	34,239,555	74,437,767
Processing	dollars	1,297,312,329	6.74	-	-	79,616,250	84,924,000	84,924,000	84,924,000	84,924,000	84,924,000	84,924,000	84,924,000	84,924,000	84,924,000
G&A	dollars	369,354,560	1.92	Owners Cost		17,980,800	18,291,380	18,739,200	18,186,400	18,742,440	20,124,520	19,780,300	21,332,640	21,333,360	20,988,520
Concentrate Trucking	dollars	60,744,138		0	0	2,445,993	3,295,503	3,524,964	3,270,010	3,936,412	4,574,744	4,512,556	4,463,403	4,206,313	4,706,242
Port Costs	dollars	0													
Shipping to Smelter	dollars	0	-												
Subtotal Operating	dollars	4,443,002,184		0	0	292,011,865	308,426,354	311,429,288	299,303,690	288,144,689	260,699,595	223,793,813	243,215,024	266,654,393	305,644,585
Capital Cost															
Open Pit Mining	dollars	109,873,619		5,451,240	97,686,558	509,175	178,271	1,282,907	136,873	663,271	136,873	1,282,907	178,271	136,873	136,873
Underground Mining	dollars	559,682,584		0	0	0	0	0	0	0	55,397,418	87,759,336	111,821,697	84,215,859	21,360,906
Processing	dollars	216,750,000		63,750,000	127,500,000	21,250,000	212,500	212,500	212,500	212,500	212,500	212,500	212,500	212,500	212,500
Infrastructure	dollars	64,310,297		18,398,590	23,663,750	4,181,250	1,260,000	1,260,000	1,260,000	1,260,000	1,260,000	1,260,000	1,260,000	1,260,000	1,260,000
Environment Costs	dollars	25,000,000		0	0	0	0	0	0	0	0	0	0	0	0
Indirect	dollars	72,627,379		24,882,093	39,588,481	5,446,799	189,000	189,000	189,000	189,000	189,000	189,000	189,000	189,000	189,000
Contingency	dollars	83,845,550		16,686,073	31,917,887	5,355,982	126,000	126,000	126,000	126,000	2,657,805	4,136,828	5,236,540	3,974,882	1,102,248
Subtotal Capital	dollars	1,132,089,429		129,167,995	320,356,676	36,743,206	1,965,771	3,070,407	1,924,373	2,450,771	59,853,596	94,840,571	118,898,008	89,989,114	24,261,528
Revenue (after smelting, refining, payables, etc)			$Cdn/DMT												
Copper Concentrate															
Gold	dollars	5,018,394,451	$ 6,345	-	-	235,201,402	252,657,491	254,689,516	300,527,111	392,586,946	309,622,298	276,775,212	253,102,320	289,947,997	294,875,626
Copper	dollars	984,660,252	$ 1,245	-	-	49,012,452	65,436,488	60,770,841	62,717,346	80,833,046	58,494,316	52,853,584	50,750,328	56,582,857	68,352,547
Silver	dollars	38,805,228	$ 49	-	-	2,653,165	3,398,735	3,328,861	3,477,426	3,156,805	2,652,681	2,620,735	2,143,897	2,445,110	2,373,404
Subtotal Copper Concentrate Revenue	dollars	6,041,859,931	$ 7,639	-	-	286,867,018	321,492,714	318,789,218	366,721,883	476,576,797	370,769,295	332,249,532	305,996,545	348,975,963	365,601,576
Gold Dore															
Gold	dollars	2,582,418,083		-	-	122,687,426	131,140,848	130,675,107	154,527,496	202,352,980	160,097,252	142,799,059	130,016,849	148,857,845	151,454,717
Royalties															
Gold	dollars	266,028,439		-	-	12,526,109	13,432,942	13,487,762	15,926,911	20,822,897	16,440,184	14,685,099	13,409,171	15,358,204	15,621,562
Copper	dollars	34,463,109		-	-	1,715,436	2,290,277	2,126,979	2,195,107	2,829,157	2,047,301	1,849,875	1,776,261	1,980,400	2,392,339
Silver	dollars	1,358,183		-	-	92,861	118,956	116,510	121,710	110,488	92,844	91,726	75,036	85,579	83,069
Total Royalties	dollars	301,849,730		-	-	14,334,406	15,842,175	15,731,251	18,243,728	23,762,542	18,580,329	16,626,701	15,260,469	17,424,183	18,096,970
Total Project Revenue	**dollars**	**$ 8,322,428,284**		-	-	**395,220,039**	**436,791,387**	**433,733,073**	**503,005,650**	**655,167,234**	**512,286,217**	**458,421,890**	**420,752,925**	**480,409,624**	**498,959,323**
Pre-Tax Cashflow															
Operating Cost	dollars	4,443,002,184		-	-	292,011,865	308,426,354	311,429,288	299,303,690	288,144,689	260,699,595	223,793,813	243,215,024	266,654,393	305,644,585
Capital Cost	dollars	1,132,089,429		129,167,995	320,356,676	36,743,206	1,965,771	3,070,407	1,924,373	2,450,771	59,853,596	94,840,571	118,898,008	89,989,114	24,261,528
Revenue	dollars	8,322,428,284		-	-	395,220,039	436,791,387	433,733,073	503,005,650	655,167,234	512,286,217	458,421,890	420,752,925	480,409,624	498,959,323
Pre-Tax Cashflow	dollars	2,747,336,671		-129,167,995	-320,356,676	66,464,968	126,399,261	119,233,379	201,777,587	364,571,774	191,733,026	139,787,506	58,639,893	123,766,117	169,053,211
Pre-Tax Cumulative Cashflow	dollars			-129,167,995	-449,524,672	-383,059,704	-256,660,442	-137,427,064	64,350,524	428,922,298	620,655,323	760,442,829	819,082,722	942,848,839	1,111,902,050
Post-Tax Cashflow															
Quebec Taxes	dollars	706,687,328		-	-	716,000	5,134,568	15,477,237	34,039,261	95,734,409	49,326,256	38,088,972	17,649,688	32,341,743	38,639,169
Federal Taxes	dollars	332,150,867		-	-	-	-	-	6,086,772	40,827,792	27,492,189	23,724,071	14,363,023	17,176,677	15,332,073
Total Tax	dollars	1,038,838,195	37.8%	-	-	716,000	5,134,568	15,477,237	40,126,033	136,562,201	76,818,445	61,813,044	32,012,711	49,518,420	53,971,242
Post-Tax Cashflow	dollars	1,708,498,475		-129,167,995	-320,356,676	65,748,968	121,264,694	103,756,142	161,651,555	228,009,573	114,914,581	77,974,462	26,627,182	74,247,697	115,081,969
Post-Tax Cumulative Cashflow	dollars			-129,167,995	-449,524,672	-383,775,704	-262,511,011	-158,754,869	2,896,686	230,906,259	345,820,839	423,795,302	450,422,483	524,670,180	639,752,149
		Pre-Tax	**Post Tax**												
NPV (millions) @	**5%**	$1,311	$778												
Payback Period	**years**	3.7	4.0												
Mine Life	**years**	21.1													
IRR	**%**	29.6%	22.9%												

Table 22-16: Detailed Cash Flow – Cost and Revenue Calculations (Year 11 – Year 22)($CDN)

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
Open Pit		150,163,747													
Mill Feed															
Feed to Mill	tonnes	136,780,780		6,253,416	8,409,237	9,315,001	1,830,105	-	-	-	-	-	-	-	-
Feed to Stockpile	tonnes	13,382,967		-	291,867	-	-	-	-	-	-	-	-	-	-
Stockpile to Mill	tonnes	13,382,967		3,061,583	905,756	-	639,148	-	-	-	-	-	-	-	-
Waste	tonnes	591,085,219		29,177,805	20,232,553	8,547,993	703,699	-	-	-	-	-	-	-	-
Total Material	tonnes	754,631,933		38,492,804	29,839,414	17,862,994	3,172,952	-	-	-	-	-	-	-	-
Strip Ratio		3.94		3.13	2.17	0.92	0.28	-	-	-	-	-	-	-	-
Underground															
Feed to Mill	tonnes	42,315,827		3,285,001	3,285,007	3,284,999	3,285,000	3,285,000	3,285,000	3,285,000	3,285,000	3,285,000	3,285,001	3,246,700	756,121
Operating Cost			$/t feed												
Open Pit Mining	dollars	1,895,501,838	12.62	104,962,180	91,930,535	69,620,023	16,711,078	0	0	0	0	0	0	0	0
Underground Mining	dollars	820,089,318	19.38	70,318,509	63,288,058	64,478,886	56,673,615	56,515,805	60,090,779	71,889,707	68,396,558	68,270,093	61,922,753	56,628,457	12,938,777
Processing	dollars	1,297,312,329	6.74	84,924,000	84,924,000	84,924,000	38,783,668	22,140,897	22,140,898	22,140,903	22,140,899	22,140,901	22,140,904	21,882,759	5,096,256
G&A	dollars	369,354,560	1.92	19,741,980	19,465,860	18,481,500	16,941,860	12,767,100	12,731,200	12,731,200	12,663,400	12,623,900	12,420,000	11,897,700	11,439,300
Concentrate Trucking	dollars	60,744,138		4,705,234	4,540,341	4,395,353	1,897,470	965,217	963,557	875,771	896,666	870,555	758,634	782,756	156,443
Port Costs	dollars	0													
Shipping to Smelter	dollars	0	-												
Subtotal Operating	dollars	4,443,002,184		284,651,903	264,148,794	241,849,761	131,007,686	92,389,019	95,926,433	107,637,582	104,097,523	103,905,448	97,242,292	91,191,672	29,630,776
Capital Cost															
Open Pit Mining	dollars	109,873,619		1,682,907	136,873	136,873	136,873	0	0	0	0	0	0	0	0
Underground Mining	dollars	559,682,584		25,730,241	38,112,654	23,448,857	34,163,248	30,534,506	26,321,542	5,321,871	5,064,281	8,896,101	1,534,066	0	0
Processing	dollars	216,750,000		212,500	212,500	212,500	212,500	212,500	212,500	212,500	212,500	212,500	212,500	212,500	0
Infrastructure	dollars	64,310,297		1,260,000	1,260,000	1,260,000	575,425	328,500	328,500	328,500	328,500	328,500	328,500	324,670	75,612
Environment Costs	dollars	25,000,000		0	0	6,250,000	6,250,000	6,250,000	0	0	0	0	0	0	6,250,000
Indirect	dollars	72,627,379		189,000	189,000	189,000	86,314	49,275	49,275	49,275	49,275	49,275	49,275	48,701	11,342
Contingency	dollars	83,845,550		1,301,938	1,867,847	1,822,673	2,243,891	2,053,356	1,235,812	276,073	264,301	439,425	102,961	32,467	632,561
Subtotal Capital	dollars	1,132,089,429		30,376,586	41,778,873	33,319,904	43,668,251	39,428,137	28,147,629	6,188,219	5,918,857	9,925,801	2,227,301	618,338	6,969,515
Revenue (after smelting, refining, payables, etc)			$Cdn/DMT												
Copper Concentrate															
Gold	dollars	5,018,394,451	$ 6,345	286,488,450	313,983,450	340,798,921	192,067,345	119,755,847	121,232,559	132,712,203	139,190,265	156,738,848	148,970,561	170,078,098	36,441,990
Copper	dollars	984,660,252	$ 1,245	69,886,932	65,857,120	63,111,143	35,281,362	22,279,904	22,241,584	20,215,251	20,697,570	20,094,836	17,511,406	18,068,158	3,611,146
Silver	dollars	38,805,228	$ 49	2,547,682	2,830,881	2,733,698	1,366,315	496,498	462,249	-	117,091	-	-	-	-
Subtotal Copper Concentrate Revenue	dollars	6,041,859,931	$ 7,639	358,923,065	382,621,451	406,643,757	228,715,023	142,532,249	143,936,393	152,927,464	160,004,927	176,833,684	166,481,967	188,146,286	40,053,136
Gold Dore															
Gold	dollars	2,582,418,083		147,260,480	161,151,662	174,855,786	98,575,805	61,443,865	62,201,530	68,091,461	71,415,200	80,418,959	76,433,237	87,263,007	18,697,514
Royalties															
Gold	dollars	266,028,439		15,181,213	16,627,979	18,047,915	10,172,510	6,341,990	6,420,193	7,028,128	7,371,191	8,300,523	7,889,133	9,006,939	1,929,883
Copper	dollars	34,463,109		2,446,043	2,304,999	2,208,890	1,234,848	779,797	778,455	707,534	724,415	703,319	612,899	632,387	126,390
Silver	dollars	1,358,183		89,169	99,081	95,679	47,821	17,377	16,179	-	4,098	-	-	-	-
Total Royalties	dollars	301,849,730		17,716,424	19,032,059	20,352,484	11,455,179	7,139,164	7,214,827	7,735,662	8,099,704	9,003,842	8,502,032	9,639,325	2,056,273
Total Project Revenue	**dollars**	**$ 8,322,428,284**		**488,467,121**	**524,741,054**	**561,147,059**	**315,835,649**	**196,836,950**	**198,923,096**	**213,283,254**	**223,320,423**	**248,248,800**	**234,413,172**	**265,769,968**	**56,694,377**
Pre-Tax Cashflow															
Operating Cost	dollars	4,443,002,184		284,651,903	264,148,794	241,849,761	131,007,686	92,389,019	95,926,433	107,637,582	104,097,523	103,905,448	97,242,292	91,191,672	29,630,776
Capital Cost	dollars	1,132,089,429		30,376,586	41,778,873	33,319,904	43,668,251	39,428,137	28,147,629	6,188,219	5,918,857	9,925,801	2,227,301	618,338	6,969,515
Revenue	dollars	8,322,428,284		488,467,121	524,741,054	561,147,059	315,835,649	196,836,950	198,923,096	213,283,254	223,320,423	248,248,800	234,413,172	265,769,968	56,694,377
Pre-Tax Cashflow	dollars	2,747,336,671		173,438,631	218,813,386	285,977,394	141,159,712	65,019,794	74,849,033	99,457,452	113,304,043	134,417,551	134,943,578	173,959,959	20,094,086
Pre-Tax Cumulative Cashflow	dollars			1,285,340,681	1,504,154,068	1,790,131,462	1,931,291,174	1,996,310,968	2,071,160,001	2,170,617,454	2,283,921,496	2,418,339,047	2,553,282,625	2,727,242,585	2,747,336,671
Post-Tax Cashflow															
Quebec Taxes	dollars	706,687,328		41,699,319	57,683,887	73,933,833	34,786,596	12,890,174	14,787,609	17,466,706	22,425,667	30,689,453	29,798,972	42,449,201	928,609
Federal Taxes	dollars	332,150,867		18,324,223	25,927,750	33,024,641	17,116,463	7,651,234	8,867,068	10,248,522	12,775,675	16,308,773	15,843,547	20,673,758	386,615
Total Tax	dollars	1,038,838,195	37.8%	60,023,542	83,611,637	106,958,474	51,903,059	20,541,408	23,654,677	27,715,228	35,201,342	46,998,226	45,642,519	63,122,959	1,315,223
Post-Tax Cashflow	dollars	1,708,498,475		113,415,089	135,201,750	179,018,920	89,256,653	44,478,386	51,194,356	71,742,224	78,102,701	87,419,325	89,301,059	110,837,000	18,778,863
Post-Tax Cumulative Cashflow	dollars			753,167,239	888,368,988	1,067,387,909	1,156,644,562	1,201,122,947	1,252,317,304	1,324,059,528	1,402,162,229	1,489,581,554	1,578,882,613	1,689,719,613	1,708,498,475
		Pre-Tax	Post Tax												
NPV (millions) @	**5%**	$1,311	$778												
Payback Period	**years**	3.7	4.0												
Mine Life	**years**	21.1													
IRR	**%**	29.6%	22.9%												

23 ADJACENT PROPERTIES

There are no significant properties adjacent to the Troilus Gold Project.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Open Pit Only Case

In the development of the Troilus Gold Project PEA two options were examined:

1. Open Pit with Underground
2. Open Pit Only

The first option of Open Pit with Underground was selected for the PEA based on the following points:

- better PEA level economics
- reduced environmental footprint – smaller waste dumps and backfill of 87 pit
- reduced potential resource sterilization – larger waste dump in Open Pit Only covers larger area

The following discussion describes the Open Pit only option in more detail as to the reason for its exclusion in the PEA.

24.2 Mine Schedule – Open Pit Only

A preliminary mine plan scenario was evaluated with only open pit mining. Maximum mill throughput remained at 12.6 Mt per year. The main pit design change was the inclusion of a second phase in the Z87 pit and the removal of underground mining. Minor modifications were also made to the J pit phases due to the overlap with the Z87 phase 2. A summary of the phases for this scenario is shown in Table 24-1.

Table 24-1: Pit Phase Tonnages and Grades (Open Pit Only Case)

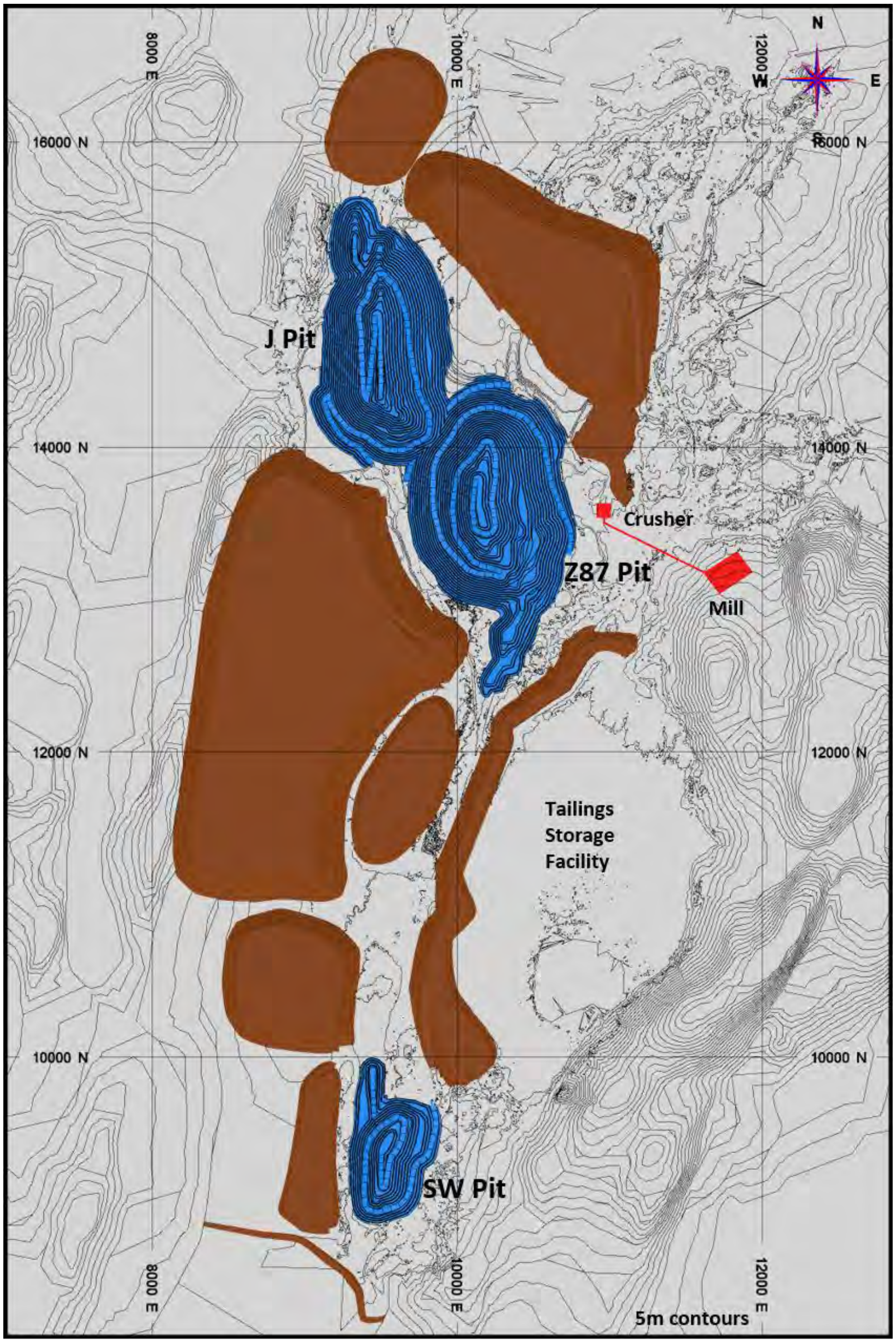
Pit	Phase	Mill Feed	Au	Cu	Ag	AuEq	Waste	Total	Strip Ratio
		(Mt)	(g/t)	(%)	(g/t)	(g/t)	(Mt)	(Mt)	
Z87	1	37.6	0.73	0.090	1.39	0.87	149	187	4.0
	2	43.2	0.71	0.083	1.38	0.84	251	294	5.8
	Subtotal	**80.8**	**0.72**	**0.09**	**1.38**	**0.85**	**400**	**481**	**5.0**
J	1	27.1	0.53	0.056	0.92	0.62	69	96	2.6
	2	28.4	0.51	0.066	0.90	0.61	103	131	3.6
	3	39.3	0.50	0.059	0.86	0.58	173	212	4.4
	Subtotal	**94.7**	**0.51**	**0.06**	**0.89**	**0.60**	**345**	**439**	**3.6**
SW	1	9.0	0.64	0.068	0.66	0.73	33	42	3.6
	2	9.8	0.65	0.062	0.85	0.74	60	70	6.2
	Subtotal	**18.8**	**0.64**	**0.065**	**0.76**	**0.74**	**93**	**112**	**5.0**
	TOTAL	**194**	**0.61**	**0.071**	**1.08**	**0.72**	**838**	**1,032**	**4.3**

An additional 247 Mt of waste for this scenario results in larger waste storage areas as shown in Table 24-2. The revised footprint of the phases and waste storage facilities is displayed in Figure 24-1. The largest change to waste storage facilities occurred with the J-waste dump (WD) as the majority of the Z87 phase 2 was sent to it. J-WD is located immediately to the southwest of both J and Z87 pits.

Table 24-2: Waste Storage Facilities Summary (Open Pit Only Case)

Waste Storage Facility	Storage Capacity (Mm^3)	Top Lift Elevation (masl)
Stockpile Base	1.1	5390
Tailings Dam Buttress	19.6	5420
J Overburden	14.5	5400
Z87 Overburden	11.2	5400
Z87-WD	67.4	5460
J-WD	255.4	5470
SW Dyke	0.3	5390
SW Overburden	4.2	5400
SW-WD	30.3	5430
Total	**404.0**	

Figure 24-1: Pit and Waste Storage Layout (Open Pit Only Case)



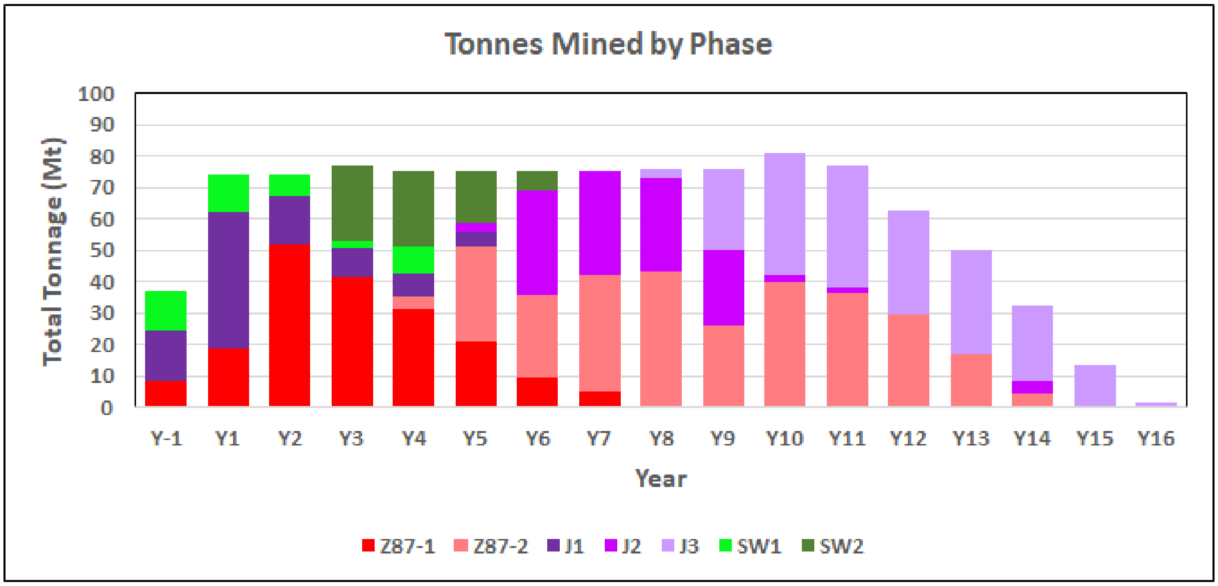
A production schedule was developed and is displayed in Table 24-3. The schedule included one pre-production year followed by 16 years mill production. No underground mining is included.

Table 24-3: Production Schedule (Open Pit Only Case)

		Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Total
Mining	**Waste (Mt)**	**34.2**	**61.4**	**61.2**	**64.5**	**59.4**	**59.6**	**61.6**	**64.2**	**67.4**	**63.3**	**71.4**	**64.1**	**47.7**	**33.8**	**17.6**	**6.0**	**0.3**	**838.0**
Summary	**Mill Feed (Mt)**	**2.8**	**12.6**	**12.6**	**12.5**	**15.6**	**15.4**	**13.4**	**10.8**	**8.6**	**12.7**	**9.6**	**12.9**	**15.1**	**16.4**	**14.7**	**7.7**	**1.0**	**194.3**
	Au (g/t)	0.47	0.48	0.51	0.53	0.64	0.70	0.76	0.72	0.45	0.54	0.51	0.54	0.69	0.73	0.67	0.58	0.60	**0.61**
	Ag (g/t)	0.66	0.81	0.99	1.15	1.12	1.21	1.29	1.19	0.98	1.01	1.07	1.09	1.20	1.10	1.07	0.92	0.94	**1.08**
	Cu (%)	0.055	0.057	0.059	0.063	0.071	0.071	0.095	0.104	0.059	0.062	0.060	0.065	0.083	0.090	0.068	0.053	0.051	**0.071**
	Total (Mt)	**37.0**	**74.0**	**73.8**	**77.0**	**75.0**	**75.0**	**75.0**	**75.0**	**76.0**	**76.0**	**81.0**	**77.0**	**62.8**	**50.2**	**32.3**	**13.7**	**1.3**	**1,032**
Processed	**Mill Feed (Mt)**	**0.0**	**11.8**	**12.6**	**12.6**	**12.6**	**12.6**	**12.6**	**12.6**	**12.6**	**12.6**	**12.6**	**12.6**	**12.6**	**12.6**	**12.6**	**12.6**	**6.1**	**194**
Material	Au (g/t)	0.00	0.52	0.51	0.53	0.69	0.77	0.78	0.68	0.43	0.54	0.48	0.55	0.75	0.85	0.72	0.49	0.40	**0.61**
	Ag (g/t)	0.00	0.82	0.99	1.15	1.26	1.32	1.31	1.11	0.87	1.01	0.97	1.09	1.28	1.18	1.12	0.87	0.82	**1.08**
	Cu (%)	0.000	0.058	0.059	0.064	0.078	0.077	0.097	0.094	0.055	0.062	0.058	0.066	0.085	0.096	0.070	0.058	0.063	**0.071**
Stockpile	**Low Grade (Mt)**	**2.4**	**3.6**	**3.6**	**3.5**	**6.3**	**9.3**	**10.0**	**8.3**	**4.2**	**4.3**	**1.3**	**1.6**	**4.1**	**7.9**	**10.0**	**5.1**	**0.0**	
Balance	Au (g/t)	0.36	0.36	0.36	0.36	0.38	0.39	0.39	0.38	0.38	0.37	0.37	0.37	0.36	0.36	0.36	0.36	0.00	
	Ag (g/t)	0.60	0.68	0.68	0.66	0.60	0.64	0.65	0.66	0.67	0.67	0.66	0.73	0.78	0.80	0.79	0.79	0.00	
	Cu (%)	0.047	0.053	0.053	0.050	0.045	0.045	0.047	0.048	0.051	0.051	0.051	0.052	0.066	0.067	0.065	0.065	0.000	
	High Grade (Mt)	**0.5**	**0.0**	**0.0**	**0.0**	**0.2**	**0.0**	**0.1**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	
	Au (g/t)	1.05	0.00	0.00	0.00	0.90	0.00	0.84	0.00	0.90	0.90	0.90	0.90	0.90	0.86	0.86	0.86	0.86	
	Ag (g/t)	0.96	0.00	0.00	0.00	0.81	0.00	0.91	0.00	1.58	1.58	1.58	1.58	1.58	1.01	1.01	1.01	1.01	
	Cu (%)	0.095	0.000	0.000	0.000	0.041	0.000	0.042	0.000	0.088	0.088	0.088	0.088	0.088	0.044	0.044	0.044	0.044	
Stockpile Reclaim	**(Mt)**	**0.0**	**0.5**	**0.0**	**0.4**	**0.0**	**0.2**	**0.0**	**2.2**	**4.5**	**0.0**	**3.1**	**0.0**	**0.0**	**0.0**	**0.0**	**4.9**	**5.1**	**20.8**
Total Movement	**(Mt)**	**37.0**	**74.5**	**73.8**	**77.4**	**75.0**	**75.2**	**75.0**	**77.2**	**80.5**	**76.0**	**84.1**	**77.0**	**62.8**	**50.2**	**32.3**	**18.6**	**6.4**	**1,053**

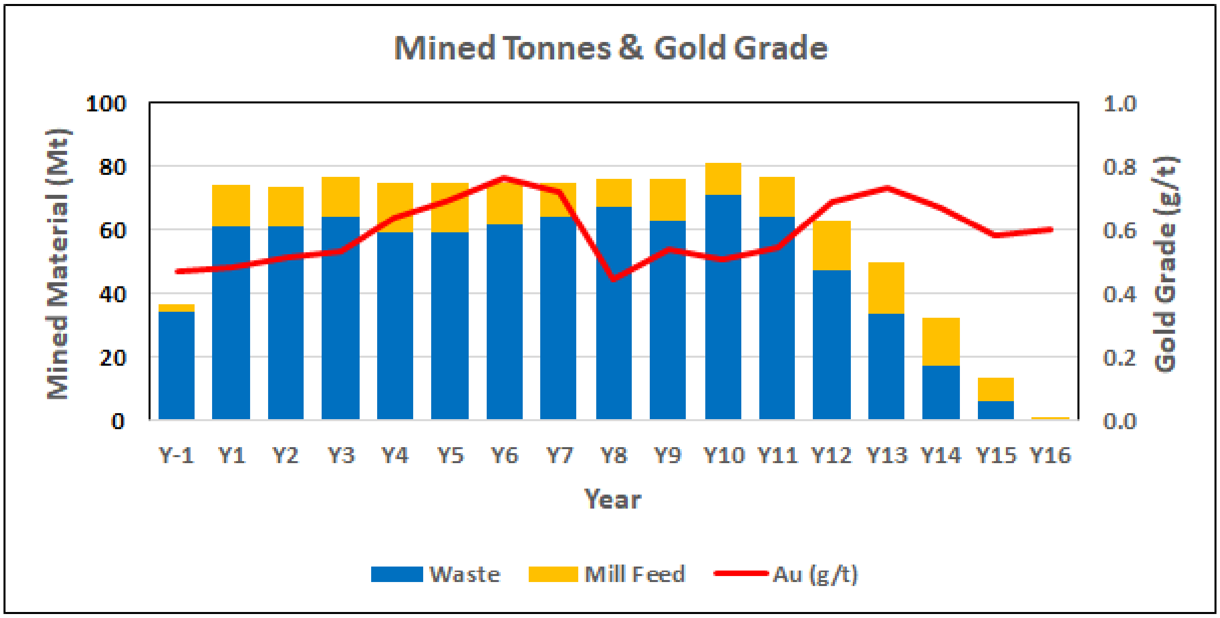
The peak mining rate of 81 Mt per year occurred in year 10 as displayed in Figure 24-2. As the Z87 phase 2 contained significant waste stripping in the upper elevations, mining was started in year 4 and continued to year 14. The SW phases were mined complete by year 6. Mining is active in J pit throughout the schedule.

Figure 24-2: Tonnes Mined by Phase (Open Pit Only Case)



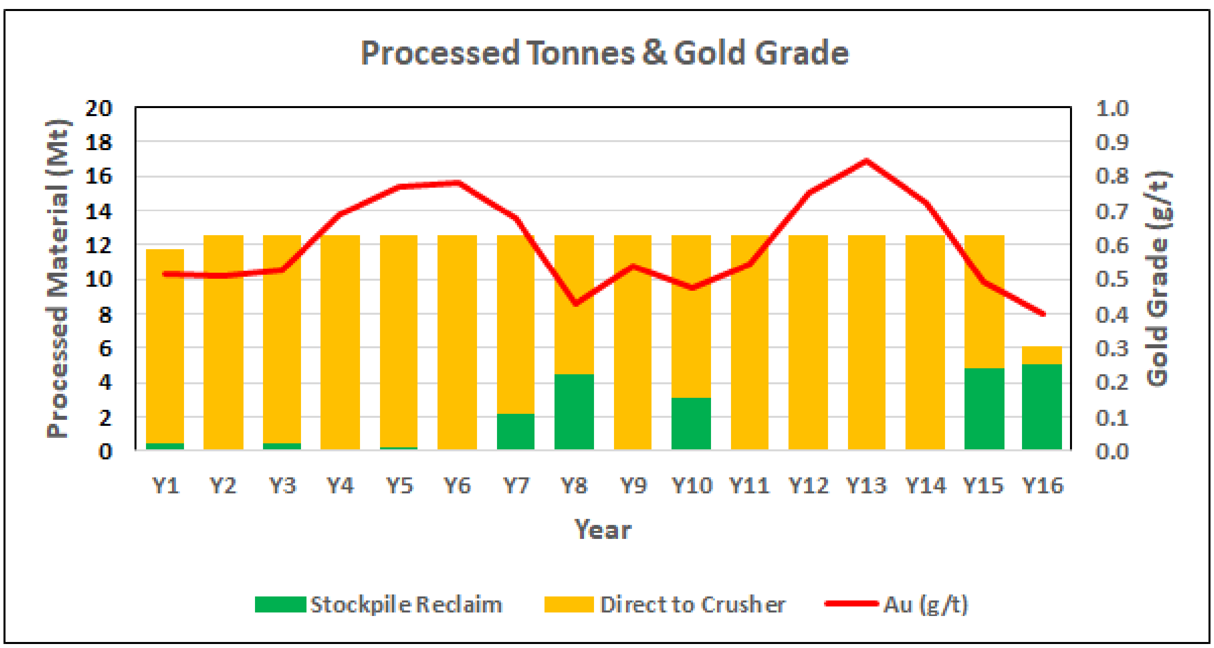
Mined material type and gold grade are shown in Figure 24-3. Waste stripping is relatively consistent for years 1 to 11. Lower gold grades are mined in years 8 to 11 from Z87 Phase 2, and J Phases 2 and 3.

Figure 24-3: Mined Tonnes & Gold Grade (Open Pit Only Case)



The process tonnages and gold grade are shown in Figure 24-4. Gold grades dip during the years 8 to 11 due to having to supplement feed with stockpiled low-grade material and lower grades being exposed in pits. The full mill capacity is achieved for years 1 to 15.

Figure 24-4: Process Tonnes & Gold Grade (Open Pit Only Case)



24.3 Capital and Operating Cost Estimate – Open Pit Only

The revised mine schedule was used to generate an updated capital and operating cost estimate for this case. This included revised hauls to properly determine the mine equipment fleet.

24.3.1 Capital Cost Estimate

The mine fleet was fully financed following the same logic as applied to the Base Case of open pit and underground. Mine equipment purchases and timing are shown in Table 24-4 and Table 24-5

Table 24-4: Open Pit Only - Equipment Purchases

Equipment	Unit Life (hrs)	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	Yr 15	Yr 16
Production Drill	25,000	4	5	1				2	6	1	2							
Production Loader	35,000	1	1									1						
Hydraulic Shovel	60,000	2	2						1		2	1						
Haulage Truck	60,000	8	13	3	3	4		1	3		10	15	1					
Crusher Loader	25,000		1								1							
Tracked Dozer	35,000	5	1							3	3							
Grader	20,000	3								3								

Table 24-5: Open Pit Only – Equipment Fleet Size

Equipment	Unit Life (hrs)	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	Yr 15	Yr 16
Production Drill	25,000	4	9	10	10	10	10	10	10	10	9	11	11	11	11	11	11	11
Production Loader	35,000	1	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1
Hydraulic Shovel	60,000	2	4	4	4	4	4	4	4	5	5	5	4	4	4	4	4	4
Haulage Truck	60,000	8	21	24	27	31	31	32	35	35	39	39	39	37	34	32	30	30
Crusher Loader	25,000		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Tracked Dozer	35,000	5	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Grader	20,000	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3

The portion of the mine equipment that is not financed is shown in Table 24-6.

Table 24-6: Open Pit Only – Mining Capital Cost Estimate ($CDN)

Equipment	Preproduction Year -2, -1 ($CDN)	Sustaining ($CDN)	Total ($CDN)
Pre-Production Stripping	88,170,000	-	88,170,000
Mining Equipment			
Major Equipment	3,238,000	4,246,000	7,484,000
Support Equipment	1,760,000	885,000	2,645,000
Subtotal – Mining Equipment	**4,998,000**	**5,131,000**	**10,129,000**
Miscellaneous Mine Capital			
Engineering Office Equipment	1,200,000	-	1,200,000
Dispatch System	1,000,000	-	1,000,000
Dewatering System	1,239,000	6,850,000	8,089,000
Subtotal – Miscellaneous Mine Capital	**3,439,000**	**6,850,000**	**10,289,000**
Total Mine Capital	**96,607,000**	**11,981,000**	**108,588,000**

The plant, infrastructure, environmental, indirects and contingency of the open pit only case are shown together with the mine capital in Table 24-7. There are minor variations in the plant and infrastructure sustaining capital costs due to the shorter mine life, but the most significant change is the elimination of the underground capital.

Table 24-7: Open Pit Only – Capital Cost Estimate

Area	Initial Capital (M$CDN)	Sustaining Capital (M$CDN)	Total Capital (M$CDN)
Open Pit – Prestrip (capitalized)	*88.2*	-	*88.2*
Open Pit - Capital	*8.4*	*12.0*	*20.4*
Open Pit Mining - Subtotal	96.6	12.0	108.6
Underground Mining	-	-	-
Processing	191.3	30.8	222.1
Infrastructure	42.1	22.4	64.5
Environmental	-	25.0	25.0
Indirects	64.4	8.1	72.5
Contingency	48.6	9.7	58.3
Total	**442.9**	**108.0**	**550.9**

24.3.2 Operating Cost Estimate

With the modified mining sequence required for the open pit only case, the open pit mining operating costs changed to reflect the longer and higher haulage routes required for the additional waste material. As well, the elimination of the 87 pit backfill also added to the haulage cost increase. The labour plus repair and maintenance unit rates per hour for the equipment remain unchanged but truck productivities were affected resulting in a higher cost per tonne moved. The support equipment factors remained the same therefore additional support equipment time was required in some instances when it was linked to haulage hours. Grading time for example increased due to the additional haulage distances and haulage hours.

The open pit only case mine operating costs are shown in Table 24-8 and Table 24-9.

Table 24-8: Open Pit Only Case – Mine Operating Costs with Financing ($CDN/t Total Material)

Open Pit Operating Category	Unit	Year 1	Year 3	Year 5	LOM Average Cost
General Mine and Engineering	$CDN/t	0.19	0.18	0.17	0.20
Drilling	$CDN/t	0.26	0.24	0.23	0.24
Blasting	$CDN/t	0.49	0.47	0.48	0.49
Loading	$CDN/t	0.17	0.20	0.18	0.18
Hauling	$CDN/t	0.72	0.90	0.97	0.88
Support	$CDN/t	0.20	0.20	0.19	0.23
Grade Control	$CDN/t	0.09	0.06	0.06	0.10
Finance Costs	$CDN/t	0.14	0.26	0.41	0.49
Dewatering	$CDN/t	0.01	0.01	0.01	0.01
Total	**$CDN/t**	**2.27**	**2.50**	**2.69**	**2.79**

Table 24-9: Open Pit Only Case – Mine Operating Costs with Financing ($CDN/t Mill Feed)

Open Pit Operating Category	Unit	Year 1	Year 3	Year 5	LOM Average Cost
General Mine and Engineering	$CDN/t mill feed	1.14	1.08	1.08	1.03
Drilling	$CDN/t mill feed	1.54	1.42	1.50	1.26
Blasting	$CDN/t mill feed	2.90	2.82	3.11	2.57
Loading	$CDN/t mill feed	1.03	1.19	1.16	0.95
Hauling	$CDN/t mill feed	4.26	5.41	6.22	4.62
Support	$CDN/t mill feed	1.20	1.20	1.20	1.20
Grade Control	$CDN/t mill feed	0.53	0.35	0.37	0.50
Finance Costs	$CDN/t mill feed	0.86	1.54	2.65	2.59
Dewatering	$CDN/t mill feed	0.03	0.03	0.03	0.03
Total	**$CDN/t mill feed**	**13.49**	**15.05**	**17.32**	**14.75**

Process operating unit costs remain the same for this case and are unaffected by the change in the open pit. The unit cost remains at $CDN 6.74 /t mill feed.

General and Administrative costs change as a result of fewer people in camp with no underground operation. The life of mine average for G&A drops to $CDN 1.50 /t mill feed or an average of $CDN 18.5 million per year. Life of mine the G&A cost is $CDN 291 million.

The concentrate trucking unit cost also remains the same at $CDN 70.30 per wet metric tonne of concentrate.

The total estimated operating cost for the open pit only case is shown in Table 24-10.

Table 24-10: Open Pit Only – Troilus Gold Project Operating Costs ($CDN)

Area	Units	Open Pit Only (Year 1 – 5)	Life of Mine (Year 1-16)
Open Pit Mining	$CDN/t moved	2.66	2.79
	$CDN/t moved	16.04	14.29
Processing	$CDN/t moved	6.74	6.74
G&A	$CDN/t moved	1.50	1.50
Concentrate Trucking	$CDN/t moved	0.26	0.29
Total Operating Cost	**$CDN/t moved**	**24.54**	**22.82**

24.4 Financial Analysis – Open Pit Only Case

The same simple pre-tax and post-tax cash flow model used for the Base Case was also used to evaluate the Open Pit Only case. This discounted cash flow (DCF) was created in Excel for Troilus use. The above mentioned operating and capital costs were used in the financial analysis in addition to the same metal pricing regime as used for the PEA Base Case.

Like the PEA Base Case, readers are cautioned that the PEA is preliminary in nature. It includes inferred mineral resources considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty the PEA will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, socio-political, marketing, or other relevant issues.

All pricing is in H2 2020 Canadian dollars unless otherwise noted.

The metallurgical recoveries and concentrate grades by zone remained the same in this case. Due to a slightly different proportion of material feed, the final grades of the concentrate for the life of the mine have changed slightly. The LOM concentrate grade is 17% copper with a gold grade of 97 g/t and silver of 114.7 g/t.

The results of the financial analysis are shown in Table 24-11.

Table 24-11: Troilus Gold Project – Open Pit Only DCF Financial Summary ($CDN)

Parameter	Units	Pre-Tax	Post-Tax
Metal Prices			
Gold	$US/oz	1,475.00	
Copper	$US/lb	3.00	
Silver	$US/oz	20.00	
Exchange Rate	$CDN:$US	0.74	
Net Present Value (5%)	$CDN M	$1,189	**$701**
Internal Rate of Return	%	28.5	**22.3**
Net Revenue less Royalties	$CDN M	7,329.5	**7,329.5**
Total Operating Cost	$CDN M	4,433.7	**4,433.7**
Life of Mine Capital Cost	$CDN M	550.9	**550.9**
Taxes	$CDN M	-	**906.3**
Net Cash Flow	$CDN M	2,344.9	**1,438.6**
Payback Period	Years	4.0	**4.2**
Cash Costs (with credits)	$CDN/oz	1,112	**1,384**
All-in Sustaining Cost	$CDN/oz	1,145	**1,417**
Payable Metals (Life of Mine)			
Gold	Moz	3.33	
Copper	M Lbs	254	
Silver	Moz	1.79	
Initial Capital	$CDN M	442.9	
Sustaining Capital	$CDN M	108.0	
Total Capital	$CDN M	550.9	
Mine Life	Years	16	

The taxation model considered the proper taxation for the jurisdiction of Quebec 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 in the same manner as the PEA base case.

The LOM taxes determined were:

1. Quebec Taxes = $CDN 631.3 million
2. Federal Taxes = $CDN 275.1 million
3. Total Taxes = $CDN 906.4 million

The amount of tax is approximately 38.7% of the pre-tax cash flow. Taxation added 0.2 years to the payback period of the project.

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
- royalties set at 3.5% for all metals per current Troilus Gold agreements
- all capital expenditures are depreciated depending on the class of asset.

For those accustomed to comparing projects in United States dollars, Table 24-12 has been prepared using an exchange rate of 0.74 ($CDN:$US).

Table 24-12: Troilus Gold Project – Open Pit Only DCF Financial Summary ($USD)

Parameter	Units	Pre-Tax	Post-Tax
Metal Prices			
Gold	$US/oz	1,475.00	
Copper	$US/lb	3.00	
Silver	$US/oz	20.00	
Exchange Rate	$CDN:$US	0.74	
Net Present Value (5%)	$M	881	**520**
Internal Rate of Return	%	29.6	**22.9**
Net Revenue less Royalties	$USD M	5,429	**5,429**
Total Operating Cost	$USD M	3,284	**3,284**
Life of Mine Capital Cost	$USD M	408	**408**
Taxes	$USD M	-	**671**
Net Cash Flow	$USD M	1,737	**1,066**
Payback Period	Years	3.7	**4.0**
Cash Costs (with credits)	$USD/oz	824	**1,025**
All-in Sustaining Cost	$USD/oz	848	**1,049**
Payable Metals (Life of Mine)			
Gold	Moz	3.84	
Copper	M Lbs	265	
Silver	Moz	1.47	
Initial Capital	$USD M	328.1	
Sustaining Capital	$USD M	80.0	
Total Capital	$USD M	408.1	
Mine Life	Years	16	

25 INTERPRETATION AND CONCLUSIONS

25.1 Geology

The Troilus Gold Project is made up of three main mineralized zones: Z87 Zone, J4/J5 Zone, and the SW Zone. The Z87 Zone and 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 SW Zone, situated approximately 2 km southwest of the Z87 Zone and was the subject of a recent drill program, was found to contain considerable mineralization and a preliminary mineral resource has been established. The gold grades within the interpreted mineralized zones are continuous and may still be open along strike and at depth.

The main mineralized zones on the Property occur around the margins of the Troilus Diorite, and comprise the Z87 Zone and the J4/J5 Zone. The SW Zone lies along strike of the Z87 Zone and may comprise two limbs of a synclinal fold. Other important mineralization discovered to date include the northern continuity of the J4/J5 Zone, the Allongé Zone, and the southwestern margin of the metadiorite (including the Z86).

Troilus 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 February 2020, Troilus completed several diamond drill core programs which support the mineral resources along strike and at depth at the Z87 Zone and the J4/5 Zone; and supports the initial mineral resources in the 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 Troilus Project, within an optimized constraining shell, at a 0.3 gpt AuEQ cut-off grade are: Indicated resources of 164.2 Mt at 0.68 gpt Au, 0.08 %Cu, 1.20 gpt Ag and 0.80 gpt AuEQ; and Inferred resources of 101.2 Mt at 0.60 gpt Au, 0.07 %Cu, 1.12 gpt Ag and 0.70 gpt AuEQ.

The mineral resources amenable to underground mining scenario, for contiguous blocks below the optimized constraining shells, at a 0.9 gpt AuEQ cut-off grade are: Indicated resources of 13.1 Mt at 1.61 gpt Au, 0.13 %Cu, 0.91 gpt Ag and 1.79 gpt AuEQ; and Inferred resources of 15.5 Mt at 1.62 gpt Au, 0.11 %Cu, 0.52 gpt Ag and 1.77 gpt AuEQ. The effective date of the Troilus Project Mineral Resources is 20 July 2020.

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 exploration and development is warranted and recommended for the Project.

25.2 Open Pit Mining

The PEA is based on the reactivation and expansion of the Z 87 and J Zone pits and the addition of a new area, the SW Zone. These pits provide the open pit feed material necessary to maintain the process plant feed rate at 35,000 t/d while operational.

The Z 87 pit is a single phase which provides 36.6 Mt of mill feed grading 0.72 gpt gold, 0.088% copper and 1.4 gpt silver for a gold equivalent grade of 0.85 gpt. Waste from this pit totals 149.4 Mt for a strip ratio of 4.1 (waste: mill feed). The Z 87 pit forms the top of the underground development.

The J zone pit has three phases. The phases total 94.8 Mt of mill feed grading 0.51 gpt gold, 0.06% copper, and 0.89 gpt silver for a gold equivalent grade of 0.60 gpt. Waste from the phases totaled 348.7 Mt for a strip ratio of 3.7:1 (waste: mill feed).

The new SW Zone pit is mined in two phases. They will produce 18.8 Mt of mill feed grading 0.64 gpt gold, 0.065% copper and 0.76 gpt silver for a gold equivalent grade of 0.74 gpt. The waste amounts to 93 Mt giving a strip ratio of 5.0:1 (waste: mill feed).

Due to the multi-metal nature of the deposit the mill feed cut-off is based on a value per tonne which is often referred to as the milling cut-off. This is determined as:

(Block Revenue – Milling Cost – G&A Cost)/Block Tonnage

The lower cut-off used is any block with a value per tonne equal to or greater than $0.01 /t. Due to the variable grades and recoveries this ranges from 0.08 gpt gold for the Z 87 pit to 0.13 gpt gold for J pit. A higher grade cut-off of 0.7 gpt was also used to segregate the mill feed material into low and high grade and assist in the project economics.

The phases are scheduled to provide 35,000 t/d of feed to the mill over a 14 year open pit mining life after one year of pre-production stripping. As the underground mine production comes online in Year 8 the open pit production drops to a level sufficient to keep the process plant at full capacity. The pits are sequenced to minimize initial stripping and provide higher feed grades in the early years of the mine life. This is accomplished with stockpiling of lower grade material which is used later in the mine life.

Initial mining starts in the Z 87 pit and the SW pit. These provide the highest grade to the mill early in the schedule. The Z 87 pit needs to be complete for the underground mine to produce material. The 87 pit finishes in Year 6. The other advantage of finishing the Z 87 pit early is that this can then be used for waste storage of material from the J pit.

The pits are built on 10 metre benches with safety berm placement each 20 metres. Inter-ramp angles vary from 47 to 53 degrees depending upon the wall orientation. Minimum mining widths of 60 metres were maintained in the design. Ramps are at maximum 10% gradient and vary in width from 25.5 m (single lane width) to 33.2 m (double lane width). They have been designed for 181 t haulage trucks.

The mine equipment fleet is anticipated to be financed to lower capital requirements. The fleet will be comprised of nine 200mm down the hole drills, two 22 m^3 hydraulic shovels and two 23 m^3 front end loaders. The truck fleet will total 28 trucks from Year 1 onwards. This is due to the long hauls from the SW pit, the tailings buttress buildup and the initial higher strip ratio reactivating the Z 87 and J pits. The usual assortment of dozers, graders, small backhoes, and other support equipment is considered in the equipment costing. A smaller front end loader (13 m^3) will be stationed at the primary crusher.

The waste dumps will be placed adjacent to the various pits. Waste from the Z 87 pit will be used to recontour and build upon the existing Z 87 waste dump. This will include wraparounds on the eastern side which will form the base for the low grade stockpile. The SW pit will develop a new waste dump to the west of the pit. The J pits will cover over an older facility to the south until the Z 87 pit is available for backfill. When that occurs, all remaining waste will be placed in the Z 87 pit from the J phases or on the tailings buttress. This allows the reclamation of the other facilities to be completed while mine operations are underway. A total of 292.1 Mm^3 has been designed and it is sufficient for the mine needs.

The LOM operating cost is estimated at $CDN 2.70/t of material mined. This includes equipment financing of $CDN 0.41/t of material mined.

Pre-production stripping costs of $CDN 94.7 million are capitalized. Initial mine equipment capital is $8.4 million with sustaining capital of $CDN 6.7 million.

25.3 Underground Mining

Troilus plans that development of the underground mine will commence once open pit production is established. Underground production will be mined concurrently with lower grade open pit material, thereby enhancing mill grade.

Inferred Resources account for 28% of the underground material to be processed. Mineral Resources that are not Mineral Reserves and do not have demonstrated economic viability. The preliminary economic assessment is preliminary in nature in that it includes Inferred Mineral Resources that are considered too speculative to have economic considerations applied to them and should not be relied upon for that purpose. Only limited underground mine planning has previously been undertaken on the Z87 deposit.

The planned underground mining area is an extension of the Z87 deposit previously mined by open pit at Troilus. The depth of the existing open pit is now planned to be extended by open pit methods by around 50 m, to approximately 350 m below surface. The currently identified Measured, Indicated and Inferred Resources for the underground area extend to around 900 m below surface and measure a maximum of approximately 850 m along strike. The dip of the deposit varies from around 60^0 to around 40^0, averaging 55^0 in the north and central areas with the flatter dip to the south. An optimised in situ cut off grade of 0.8 equivalent g/t Au was calculated. Higher grade mineralized areas bifurcate in certain areas, but low grade intervening mineralization that allows for the mining of the full section from footwall to hangingwall at satisfactory grades was included in the study plans. Stopes vary in thickness up to 80 m true thickness with the thickness generally reducing with depth.

In general, ground conditions are considered to be good to very good with strong rock throughout the footwall, orebody and hanging wall sequences. Geological structure in the form of faults and low-angle, widely spaced joints have been identified in the exposed open pit sidewalls.

Trade off studies were undertaken that identified Slot and Mass Blast (S&MB) as the preferred mining method and Rail-Veyor as the preferred materials handling system.

S&MB will be the primary underground mining method used to exploit the Z87 deposit below the open pit floor and will provide 89% of the life of mine underground feed to the mill. The remaining 11% of underground mill feed will be mined using the sub level caving (SLC) method, which is located in the upper portion of the underground mining area, between the deepened open pit and the upper-most level of slot and mass blast stopes. Both of the selected mining methods - as well as the development and operation of the Rail-Veyor materials handling system - operate in a 'top-down' fashion, thus minimising and deferring the mine development necessary to place the mine in operation and sustain production over the life of mine. Initial production will be by SLC followed by S&MB.

Life of mine feed to the process plant is estimated to be 42.3Mt with an equivalent gold grade of 1.35 g/t at a steady-state production rate of 9,000 tpd. Underground mine capital expenditure during the 4.5 year pre-production period is estimated to be CDN$ 324 M with CDN$ 236 M sustaining capital during the remainder of mine life. Capital development during the pre-production period will be undertaken by a contractor. Owner crews will undertake all subsequent mine activities apart from raising and deposit delineation. Life of mine underground operating costs are estimated to average CDN$ 19.38 per tonne of process feed.

Large scale mobile equipment types were assumed to maximise productivity. The mobile equipment fleet will be leased.

Potential concerns or risks noted by AGP during the course of the study may be summarised as follows:

- Whilst S&MB is not a common mining method, it has been used successfully at large-scale mining operations in Canada and worldwide. S&MB appears to have been first used at several mines in Canada in the 1960's and 70's, and more recently in Australia and elsewhere worldwide. The very large mass pillar blasts will be technically challenging but have been successful elsewhere. Site visits for benchmarking purposes are recommended to operations using this method for primary ore extraction.
- Fragmentation in the stope will likely be somewhat variable and large blocks will likely occur. Secondary breakage at stope draw points will be required. In mitigation, AGP believes that conservative stope production rates have been scheduled.
- There may be potential for reduced broken material recovery in stopes with flatter footwall dips. This aspect may be considered in future, more detailed, stope planning.
- In future, dilution estimates should be confirmed by more rigorous analysis.

- The Rail-Veyor system is considered an innovative and emerging technology. It is a rail-based remotely-controlled haulage system which consists of a number of individual trains running on light rail track. It will be important for project and manufacturers engineers to work together to finalise system designs and ensure planned system productivity and costs. Two production systems are in operation in North America with good production track records under similar duty cycles. Site visits should be made to benchmark these operations.
- The open pit and underground mine will be connected across the full strike length from the commencement of underground production. There is potential for significant water inflows and mud-rush events. More detailed analysis of potential water inflows, dewatering requirement assumptions and system designs will be required in future.

25.4 Metallurgy and Processing

Although a straightforward grinding plus froth flotation approach has been selected as the recommended option for processing Troilus mineralization, a number of processing options have been evaluated dating back to 1991, namely:

1. coarse bottle roll cyanidation (indicative of heap leach recovery)
2. bottle roll cyanidation of milled material (Indicative of tank leach recovery)
3. gravity recovery of gold followed by flotation to produce a copper-gold concentrate
4. flotation followed by cyanidation of the flotation tails
5. bottle roll cyanidation of milled material followed by flotation of cyanidation tails

Processing options 1 and 2 recover only gold, whereas options 3 to 5 recover both copper and gold to payable streams. It should be noted that option 5, leaching followed by flotation of the leach tails is not widely practiced, and AGP knows of no commercial operations that currently adopt this strategy. However, this should not preclude further testwork and evaluation.

Heap leach amenability testing on J4 Zone, J5 Zone and 87 Zone material indicates that gold recoveries of 49%, 45% and 55% respectively could be obtained. There is some evidence to suggest that HPGR crushing ahead of coarse bottle roll testing increased gold recovery significantly, but this should be tested further, along with column leach testing over longer durations to properly evaluate the merits of heap leaching for Troilus mineralization. For completeness, a summary of the coarse bottle roll test results on all composites is given in Table 25-1 below:

Table 25-1: Summary of Coarse Bottle Roll Test Recovery (Heap Leach Amenability)

Composite	Au Head Grade, g/t	Au Recovery, %
J4 Zone	1.04	49
J5 Zone	0.83	45
87 Zone	1.14	55
Average	**1.00**	**50**

Gold recoveries increase significantly when leaching of finely ground material is employed. The average gold recoveries and head grades for J4 Zone, J5 Zone and 87 Zone milled bottle roll tests at P_{80}s of 75µm and 150µm are summarised in Table 25-2 below:

Table 25-2: Tank Leach Metallurgical Recovery Projection

Composite and P_{80}	Au Head Grade, g/t	Au Recovery, %
J4 Zone 150µm	1.38	87
J5 Zone 150µm	0.96	79
87 Zone 150µm	0.74	84
Average 150µm	**1.03**	**83**
J4 Zone 75µm	1.61	95
J5 Zone 75µm	1.00	85
87 Zone 75µm	0.84	92
Average 75µm	**1.15**	**90**

Finer primary grinding yields an increase in gold recovery of ~7%, bringing average gold recovery for all zones to 90%.

A large volume of metallurgical test data has been reviewed, dating back to the early 1990's. In preparing the metallurgical performance predictions required for economic evaluation, the following factors have been considered and form part of the methodology:

- Gravity gold recovery from the 1993 and 2003 bench scale testwork on 87 Zone and J4 Zone material (excluding the high grade pilot plant sample) averaged 40%. Concentrate grades were reported to be >800g/t Au, which is thought to be well within the range for industrial operation and quite suitable feed for intensive cyanidation and doré production. Modern methods of modelling gravity recoverable gold (GRG) content have evolved since the time of historical testwork and it is thought likely that modern E-GRG testwork at coarser grinds, representative of actual plant conditions (gravity concentration would likely be conducted on the cyclone underflow), might result in a slight de-rating of this performance point. For the purposes of this preliminary economic study, AGP has assumed that like historical Troilus operations, gravity concentration would be carried out on the primary ball mill cyclone underflow and also on the regrind mill cyclone underflow streams. A somewhat conservative gravity recovery of 30% for gold and 0% for copper has been used for the economic analysis.
- Recent flotation testwork at KCA only involved the rougher portion of the circuit (i.e.: no cleaner circuit data). Gravity concentration, rougher concentrate regrinding and cleaner flotation was not included and therefore projections of final concentrate grade and overall recoveries have been estimated using average cleaner circuit performance in other tests, equivalent to 95% per stage of cleaning.
- Good quality full flowsheet testwork dates from 1993 and 2003 and although this work was completed to a high standard by a reputable laboratory, the samples being tested at the time were likely taken from zones within the mine plans of that time (1996 to 2010). As these zones are now mined out, the composition of the samples used in the early work

could differ in grade, mineralogy and metallurgical performance from the mineralization selected in the current preliminary mine plans.

A range of grades and composites from the J4 and 87 Zones were tested, and the following inferences can be made that would require supporting testwork on representative samples from future zones of production:

- Conventional milling and flotation flowsheets have been successfully tested on Troilus composites as part of the 1993 Kilborn feasibility study. Primary grinding with P_{80} between 85 and 100µm was employed during pilot plant testwork, followed by rougher flotation, regrinding and three stages of copper flotation. Gravity concentration was employed in the primary and regrind circuits. Copper and gold recoveries were 90% and 88% respectively, based on a 1.95g/t Au and 0.16% Cu head grade. Final flotation concentrate graded 15% Cu.
- Bench scale flotation testwork using a similar process flowsheet to the pilot plant but on lower head grade material (0.05% Cu to 0.11% Cu) returned lower concentrate grades and copper recoveries (62-85% copper recovery at 4 to 11% Cu concentrate grades) while combined gravity plus flotation gold recoveries ranged from 87 to 94%. The lower concentrate grades were attributed to pyrite dilution and a follow up testwork program indicated that higher copper concentrate grades (17-29% Cu) could be produced at high copper recovery (85-86%). This was achieved at the expense of gold recovery but further testwork involving selective flotation, gravity concentration and cyanidation of flotation tails could lead to increased gold recovery while maintaining high copper concentrate grades.

25.5 Infrastructure and Site Layout

The Troilus Gold project benefits from infrastructure remaining from the previous mine operation. This includes:

- existing tailings facility – easy to re-establish
- power line to site – existing 161 kV line with transformers and substation
- camp area – leveled pad with easily connectable services
- explosives storage area – easy to re-establish

The infrastructure and site layout take into consideration Troilus' efforts to minimize disturbance to the area and commitment for environmental stewardship.

The site infrastructure includes major items for the project such as:

- tailings facility and buttress
- process plant and primary crusher
- camp facilities
- waste rock storage facilities
- mine shop facilities

- realigned main access road
- No Name Creek diversion ditch realignment

Tailings from the process plant will be pumped as a thickened tails into the existing facility. This facility will be built in a center line construction manner with material from the open pits. Extra material will be place outside the perimeter of the tailings facility to buttress the tailings embankment. Water from the tailings will be reclaimed and pumped back to the process plant.

The new 35,000 tpd process plant will be located adjacent to the tailings facility and the transformer substation. This location provides sufficient clearance from the pit blasting activities. The primary crusher will be located to the north of the plant in the location used by the previous operation for their primary crusher.

Camp facilities will be constructed in the same area as the past camp facilities. Access to potable water, power and sewer is already in place and just needs to be reconnected. The area is already level for camp use. The facilities will be constructed by the catering company as part of their agreement.

Waste rock and overburden will be stored in separate facilities. The overburden will be used for later reclamation of the waste storage facilities. The existing Z 87 waste dump will be reactivated and recontoured as part of its use during the open pit mine operation. The J pit phases will recontour the old J pit waste dumps and build upon an area to the south of the J pit. When the Z 87 pit is finished, material from the J pit phases will be backfilled into the Z 87 pit to reduce the environmental footprint of the project. The SW pit will establish a new waste dump to the west of the proposed pit. Waste from all three pits will be used to build lifts for the tailings facility and act as a buttress.

Mine shop facilities are to be located to the north of the process plant with easy access to the pit. It will be a 10 bay shop with wash bay, tire bay and welding bay. A mine warehouse will be attached to the facility.

The existing mine access road will have to be diverted to the east and around the tailings facility prior to operation. The SW pit will interfere with the existing alignment. This realignment will ensure light vehicle traffic cannot inadvertently entire the active mining area.

The No Name Creek diversion will be realigned to avoid the expanded pits and waste storage facilities. The beginning of the realignment will be a small dike to divert water to the start of the ditch and also provide additional habitat for potential fisheries.

25.6 Economic Analysis

The PEA is preliminary in nature. It includes inferred mineral resources considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty the PEA will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, socio-political, marketing, or other relevant issues.

The life of mine capital cost for the project is estimated at $CDN 1,132M ($US 838.6M), with an initial capital expenditure of $CDN 449.5M ($US 333M) which includes $CDN 94.7($US 70.1M) pre-strip costs which are capitalized.

At a gold price of $US 1,475/oz, copper price of $US 3.00/lb, silver price of $US 20/oz and an exchange rate of 0.74 (CDN:US) the project is estimated to have an after-tax IRR of 22.9% and a pay-back period of 4.0 years after start of production. At a discount rate of 5%, the after tax NPV is estimated at $CDN 778M ($US 576M).

The cash flow model has been based on a two-year project development period, and that concentrate production commences in month one of Year 1.

Provision has been made for Quebec and Federal taxation. This represents approximately 37.8% of the pre-tax cash flow. Royalties are 3.5% each for gold, copper, and silver.

The cashflow model assumes full equity financing with no provision for interest on the cost of capital.

The project is most sensitive to changes in metal prices, and exchange rate.

26 RECOMMENDATIONS

26.1 Introduction

The Troilus Gold PEA Study has indicated a positive project. AGP recommends that Troilus Gold proceed forward with additional studies including a Prefeasibility Study (PFS). The recommendations and associated budgets by area are described further in the sections below.

A summary of the expected study costs is shown in Table 26-1.

Table 26-1: Recommended Study Budget

Area of Study	Approximate Cost ($CDN)
Geology	$13,600,000
Geotechnical	$1,261,250
Underground Mining	$100,000
Metallurgy	$500,000
Infrastructure	$500,000
Environmental	$300,000
Prefeasibility Study	$3,000,000
TOTAL	**$19,261,250**

26.2 Geology

26.2.1 Z87 Zone - J4/J5 Zone

It is recommended that continued delineation drilling continue at the Z87 and J4/J5 Zones; specifically, within the area between the two zones, at depth and along strike at the Z87 and J4/J5 Zones. Current interpretations indicate a continuity of mineralization between the Z87 and J4/J5 Zone that has not been fully investigated in the past. Additional delineation drilling will also upgrade current Inferred Resources and investigate the continuity of mineralization at depth. Approximately 19,000 m of drilling is proposed for the Z87 Zone and the area between the two zones (between 40-45 drill holes); and approximately 25,000 m of drilling for the J4/J5 Zone, mainly at depth (between 45-50 drill holes).

It is also 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 will be important for an underground mining scenario.

26.2.2 SW Zone

It is recommended further drilling continue at the SW Zone. The deposit seems to show continuity of mineralization along strike of both limbs of the interpreted synclinal fold. Both infill and delineation drilling is expected to upgrade the resources to an Indicated category. Approximately 16,000 m of drilling is proposed, between 55 - 60 drill holes.

26.2.3 Additional Recommendations

The collection of density measurements is recommended to continue on all future drill programs. Additionally, density measurements is recommended to be collected from the initial 2018 drill holes in the Z87 Zone. Density values, acquired in the southern end of the Z87 Zone in 2019, should be included in the next updated of mineral resources.

With regards to interpreted mineralized domains in the Z87 and J4/J5 Zones, some of these domains show a 'saw-tooth' along the lateral boundaries. It is recommended that further refining of the domains be carried out to remove these shapes to better represent the trend of the mineralization. As of the writing of this report, Troilus was undertaking this review at the J4/J5 Zone.

26.2.4 Advanced Studies

At the time of writing, the Project was subject of a PEA study by AGP. In the summer of 2020, metallurgical testwork and geotechnical drilling were also being carried out in support of a possible pre-feasibility study. AGP has reviewed these programs and concur that these studies are appropriate for any advanced studies on the Project.

26.2.5 Geology Estimated Budget

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 $CDN 13.6 million.

Table 26-2 presents an estimated budget of the proposed exploration and development work.

Table 26-2 Estimated Budget of Proposed Work

Proposed Work		Approximate Cost ($CDN)
Z87 and J4/J5 Zones		
Diamond Drilling (~45,000 m)	$200/m	$ 9,000,000
Re-analysis (Z87 Ag analysis, bulk density); ~ 4,000 samples	$50/sample	$ 200,000
SW Zone		
Diamond Drilling (16,000 m)	$200/m	$ 3,200,000
Subtotal		$12,400,000
Contingency		$ 1,200,000
TOTAL		**$ 13,600,000**

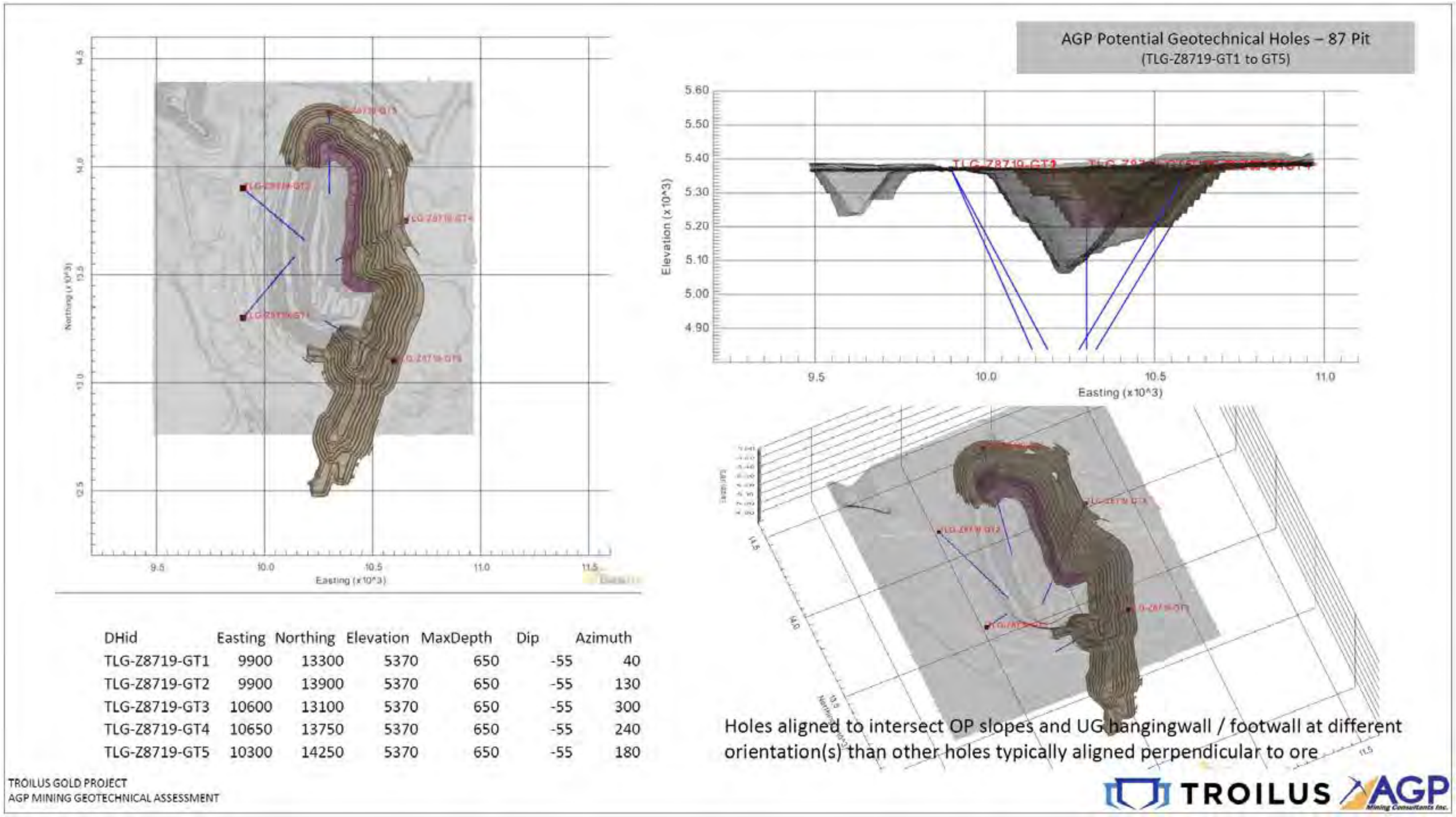
26.3 Geotechnical

AGP recognizes the potential to optimize relatively conservative initial guidance, if/when additional confirmatory data becomes available. A program of geotechnical drilling, logging, in-hole testing, field-index testing, and laboratory strength testing is required to advance the project to a PFS level. Necessary components identified in a geotechnical data-gap analysis include the following:

- Limited detailed geotechnical data has been collected to date within the pit and underground mining zones; drill holes do not intersect large portions of the proposed expanded pit walls or underground mine areas. Detailed geotechnical data, and structural data, is required for all slope sectors
- Limited spatial and geotechnical knowledge exists regarding location and intensity of fault impacted zones. Structural characterization work completed by SRK (2018) identified additional work requirements. Limited discontinuity character, orientation, and persistence data are available; these can be evaluated using targeted geotechnical coring and / or ATV/OTV surveys.
- Limited rock strength estimates exist for the project, mainly from past open pit mining studies; Further UCS, tri-axial, tensile, direct shear and other standard laboratory tests are required to determine / confirm rock strength & deformation parameters, discontinuity strength criteria.
- Conceptual PEA inflow estimates have been determined from first principles and basic modeling; these will need to be investigated directly in the field and designs updated at the next level of study.

Potential geotechnical and hydrogeological drill hole alignments and meterage required to achieve PFS-level designs for 87 Pit are illustrated below. Similar conceptual programs have also been designed for J4 and SW zones.

Figure 26-1: Proposed 87 Pit Geotechnical Holes



DHid	Easting	Northing	Elevation	MaxDepth	Dip	Azimuth
TLG-Z8719-GT1	9900	13300	5370	650	-55	40
TLG-Z8719-GT2	9900	13900	5370	650	-55	130
TLG-Z8719-GT3	10600	13100	5370	650	-55	300
TLG-Z8719-GT4	10650	13750	5370	650	-55	240
TLG-Z8719-GT5	10300	14250	5370	650	-55	180

Numerous assumptions had to be made about the primary controls on rock mass stability for open pit and underground mine designs, including assumptions regarding geology, rock mass strengths, stress, groundwater pressures, and potential failure mechanisms. As such, the current stability assessments should be considered conceptual in nature. Updated assessments that address these uncertainties are required at the next level of study. Engineering geology interpretations presented in this report should be considered preliminary. Data collected to date may not accurately reflect the rock mass comprising the final open pit walls or underground excavations. Where appropriate, geological features identified should be verified and validated with additional field work and interpretation. Additionally, seismic loading and (unlikely) multi-bench-scale to pit-scale geologic structures have been recognized as having the potential to significantly affect overall pit slope stability. The current status and impact of these are both largely unknown and will also need to be determined at the next level of study.

Empirical conceptual stability analyses and stress modeling described herein indicates S+MB is a plausible methodology at Troilus, with acknowledged risks and uncertainties, and the understanding that further work will be required to verify the concepts and designs presented. This assessment is not intended to be a “conservative” or “low-risk” assessment. AGP has assessed that the uneconomic hanging wall rock mass will generally be too strong to cave predictably and in substantial tonnages during local drawdown, particularly with stope walls fully confined by blasted ore. This requires further investigation at the next level of study. Draw control, and stress and mine sequence management thus become key controlling geotechnical parameters for mining execution.

AGP’s preliminary stress analyses of the application of S+MB method at Troilus have highlighted predictable areas of interest including stress concentrations and zones of stress shedding and relaxation, along with potentially adverse conditions in crown pillars and top drives after slot blasts, and multi-level / late-stage de-stressing of hanging walls; however, for the purposes of this PEA, the models generally exhibit stability conditions demonstrating a “reasonable prospect for extraction”.

The risk of encountering adverse features and stress conditions increases with the size of the mining-disturbed rock mass and mining depth. Considering the dimensions contemplated at Troilus, it is reasonable to expect that at least some adverse conditions will be encountered on each stoping level. Work should continue to characterize the rock mass as accurately as practical. Of importance will be the detection and precise delineation underground of faults, dykes, zones of weaker rock and any other weakening feature in the rock mass.

Hydrogeologic conditions and conceptual inflow estimates will need to be reviewed once project specific hydrogeological data is collected, as required at the next level of study.

The following items are recommended to advance the geotechnical information to the level of Prefeasibility:

1. Geotechnical Drilling, Laboratory and Fieldwork, including:
    - drilling – 5,050 m of drilling at $200/m = $1,010,000
    - geotechnical logging - $126,250
    - downhole testing - $100,000
    - laboratory testwork - $25,000
    - verify and validate current geologic features
2. Seismic Study

- o determine the seismic loading and apply to updated geologic structures to determine stability concerns if any

3. Slot and Mass Blast Analysis
   - o evaluate performance of hangingwall rock and S+MB mining method in detailed analysis of drawdown of stopes
   - o provide guidance on drawdown rates from Slot and Mass Blast stopes which affects production rate of underground mining
   - o characterize the rock mass as part of that analysis
4. Hydrogeological Analysis
   - o collect and interpret the data
   - o provide mine engineering guidance for dewatering systems

Items 2 thru 4 are estimated at ~$125,000. The majority of this analysis and study is included in the Prefeasibility study cost estimate described later, except for the drilling program (Item 1). That cost of $1,261,250 is above this and should be included in the budget separately.

26.4 Open Pit Mining

The open pit design work benefited from the experience of the previous operation. In particular, the knowledge gained on pit slopes that exist to this day. As well, the current status of the waste dumps and their stability reaffirm the design criteria. Building on that knowledge and the work completed in the PEA, the following is recommended for advancing the open pit design work to a Prefeasibility level of study:

1. Blasting Study
   - o further evaluation of pattern sizes and powder factors is required to enhance production and productivity
2. Equipment Costs and Fleet Selection
   - o update the equipment operating and capital costs from vendors
   - o examine alternate vendors for specific equipment
   - o optimize the size of haulage trucks
     - ▪ current fleet is 28 trucks of 181 tonne class
     - ▪ further study needed to determine most cost effective size
3. Review if autonomous trucks are an opportunity
   - o further study recommended
   - o technical and social benefits concerns need to be examined
4. Ore Sampling Protocols need to be established
   - o definition of ore/waste contacts
   - o sample size selection
   - o determine if blasthole samples can supplement the proposed Reverse Circulation (RC) samples
5. Pit Electrification Optimization

- examine the placement of infrastructure to bring power into the pits for shovels, drills, and dewatering

These recommendations are typically included in the normal cost of open pit design and engineering; therefore, no additional budget is listed beyond that which is allocated for the Prefeasibility study.

26.5 Underground Mining

The current design in the PEA for the underground mining portion considers the use of sublevel caving (SLC) and slot and mass blast (S&MB) stopes. Additional detailed work will be required for the areas using these methods. The following is recommended to bring the level of study up to Prefeasibility:

1. Open Pit and Underground Interface Study
   - The two SLC stoping levels directly below the open pit floor were designed to assist in the creation of the grade blanket used to act as a buffer between the blasted mill feed material and the waste material above, such that the reporting of low grade material to the S&MB draw points would be minimised. The grade blanket could be created by simply eliminating the two SLC stoping levels and moving the S&MB stope designs up to the pit floor, and subsequently reducing the mining recovery of the first level of S&MB stopes.
   - Preliminary calculations have demonstrated that the S&MB mining cost per ounce is approximately 13% lower than the mining cost per ounce for SLC stoping, resulting in a net benefit to project economics if an 'all S&MB' stoping layout was utilised.
2. Drilling and Blasting Study
   - Detailed S&MB drill layouts were not prepared for the PEA. When considering Mineral Reserves in future planning AGP recommends that specialist drilling and blasting consultants study the requirements for S&MB. The results of this work will help improve the accuracy of drilling and blasting cost estimates, as well as estimated fragmentation of broken material in the stope and secondary breakage requirements.
   - Site visits to operations using these types of large-scale, bulk methods are recommended for benchmarking purposes.
   - The fragmentation results should also be used to assess dilution estimates in more detail and to fine-tune the design of facilities to handle potential oversize as well as the muck handling system.
3. Rail-Veyor Detailed Studies
   - The Rail-Veyor materials handling system offers a number of design options, the benefits of which have yet to be quantified or planned in detail. It will be important for the design team to collaborate with Rail-Veyor to identify the appropriate final design of the material handling system. Site visits to mines operating Rail-Veyor installations are recommended.
   - The design and implementation of the Rail-Veyor system includes the likely potential to closely support capital mine development to maximise heading advance rates. It is recommended that as part of the Rail-Veyor detailed studies this be incorporated into that work. This could include the potential to support the initial pre-production mine

development from surface to minimise or eliminate development waste haulage to surface by truck, as well as improve development advance rates. In this same way, the Rail-Veyor system could be used to lower mine development costs and improve advance rates as the mine develops to depth over the life of mine.

4. Dewatering Study
    - o During the PEA study, the requirements and design of the dewatering system were identified as a major issue and potential high capital and operating cost item due to the mining method employed. With the top down mining sequence and the top of the mine open to water inflows, control of water is a top concern. AGP recommends that the assumptions and designs related to water inflow to the mine and dewatering be re-confirmed in future work.
5. Contract Mining
    - o Contract mine development cost estimates in this study were derived from first principles by AGP. A range of quotations from selected suitable contractors should be sought to confirm cost estimates.
6. Labour Study - $100,000 cost estimated
    - o AGP recommends a local survey of labour to confirm current estimates.
    - o This labour study is common to mining, processing and general and administrative estimating.

Many of the recommendations for the underground mine design would be covered under the Prefeasibility engineering study budget mentioned later. The labour study, as it is used by various disciplines is highlighted here as a cost above what is included in that estimate. The cost is shown here for the overall budget estimate. That cost is estimated at $100,000.

26.6 Metallurgy

Additional metallurgical testwork should be completed, with a focus on samples from each deposit included within the PEA mine plan, and with a focus on lower grade sample characterisation. This work would for the most part be conducted at the laboratory scale, on representative samples of drill core. Flotation work may require larger scale primary flotation testing in order to generate sufficient rougher concentrate for adequate cleaner flotation characterisation. This may necessitate the collection of large composite samples.

The extent of the deposit would also suggest the adoption of a geometallurgical approach to the metallurgical characterisation.

The pre-feasibility metallurgical program should include the following:

- Additional comminution testwork, including crusher work index confirmation (for gyratory crusher sizing). A larger database of SMC results, for example, would assist in determining variability within and/or between the deposits and would help to ensure proper SAG mill modelling and equipment selection.
- Determination of modal mineralogy plus gold deportment.

- Gravity testwork, including Extended GRG (E-GRG) characterisation.
- Flotation testwork, using larger (10-kg) test charges to ensure sufficient metal units in locked cycle cleaner circuit evaluations. Gravity concentration of rougher concentrates would be an option, although concentrate mass requirements may limit the extent of this work. Flotation testing should allow for concentrate copper grade vs copper and gold recovery target optimized concentrate copper grades, as determined through discussion with potential smelters.
- Heap leching evaluation including rolling bottle and column leach testing.
- Determination of minor (deleterious) element concentrations in flotation concentrates.
- Additional cyanidation testing, plus flotation testing on cyanidation residues.
- Cyanide detoxification testing.
- Environmental characterisation work on tailing products. This would include ABA characterisation, metals leaching work, humidity cell
- Physical characterisation of tailing products.

These studies are estimated to cost $500,000.

26.7 Infrastructure

With the addition of several new or realigned infrastructure items over the past operation, further study will be required. These additional studies should include the following:

- No Name Creek dyke by the SW Pit
- No Name Creek diversion ditch realignment
- Tailings facility
- High Voltage Line pole realignment near tailings
- New access road alignment
- Detailed surveys of plant site, crusher, diversion ditch and new waste dump foundations

This work will also include incorporation of the previously discussed geotechnical work. These studies and surveys are estimated to cost $500,000.

26.8 Environmental

Troilus Gold has an advanced understanding of the environmental concerns at the project site from the past operations and ongoing monitoring. This level of information is currently beyond what is normally associated with a PEA study and well advanced for a Prefeasibility study.

Additional background information needs to be collected, especially regarding the creek diversion realignment, future dyke by the SW pit, the SW pit area and expansion of the tailings and potential discharge. Further study will assist in providing regulators with the required additional information necessary for permitting of the proposed project.

This additional study work will require outside support beyond the current Troilus Gold teams work. An estimate of this work is $300,000 to prepare for the Prefeasibility study.

26.9 Prefeasibility Study

The level of resource classification and historical information available at the Troilus Gold project is beneficial in reducing the cost of further studies as only updates are required in some disciplines. Completing this work and combining the results of the various disciplines of geology, geotechnical, metallurgy, mining and environmental will be the focus of the Prefeasibility study lead. This work by all the disciplines beyond the previous mentioned studies is estimated to be in the order of $2-$3 million.

26.10 Recommendations and Estimated Budget

The following information in Table 26-3 is a summary of the respective discipline recommended budgets and their total costs.

Table 26-2: Summary of Recommendation Budgets

Area of Study	Approximate Cost ($CDN)
Geology	$13,600,000
Geotechnical	$1,261,250
Underground Mining	$100,000
Metallurgy	$500,000
Infrastructure	$500,000
Environmental	$300,000
Prefeasibility Study	$3,000,000
TOTAL	**$19,261,250**

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https://www.troilusgold.com/news-and-media/news-releases/ (most recently viewed 12 June 2020)

Press Release 24 May 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 7 November 2018	J4/J5 Zone drill intercepts
Press Release 29 March 2019	J4/J5 Zone drill intercepts
Press Release 19 August 2019	Z87 Zone (South) drill intercepts
Press Release 9 October 2019	J4/J5 Zone drill intercepts
Press Release 14 November 2019	J4/J5 Zone drill intercepts
Press Release 28 January 2020	SW Zone, drill intercepts
Press Release 21 April 2020	SW Zone, drill intercepts

Quebec Mining Act. (English)

http://legisquebec.gouv.qc.ca/en/ShowDoc/cs/M-13.1

Quebec Mining Act. (French)

http://legisquebec.gouv.qc.ca/fr/ShowDoc/cs/M-13.1

Worldclimate.com website: Mistissini, Quebec

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; Quebec
https://www.yr.no/place/Canada/Quebec/Mistissini~6075492/
https://www.yr.no/place/Canada/Quebec/Mont_Wapaskouch/
https://www.yr.no/place/Canada/Quebec/Pic_Longview/ (most recently viewed 2 June 2020)

28 CERTIFICATE OF AUTHORS

28.1 Gordon Zurowski, P.Eng.

I, Gordon Zurowski of Ontario, Canada, as one of the authors of this technical report titled "Preliminary Economic Assessment of the Troilus Gold Project, Quebec, Canada" dated 14 October 2020, do hereby certify that, and make the following statements:

- 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 over 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. As a result of my experience and qualifications, I am a Qualified Person as defined in NI 43–101.
- I am responsible for Sections 1.1, 1.8 to 1.12. 1.13.2, 1.14.2, 15, 16, 17, 18, 19, 20, 21 (except 21.2.3 and 21.3.3), 22, 24, 25.2, 25.3, 25.5, 25.6, 26.1, 26.3, 26.4, 26.5, 26.7, 26.8, 26.9, and 26.10 of this technical report titled "Preliminary Economic Assessment of the Troilus Gold Project, Quebec, Canada", and dated 14 October 2020.
- I have had no prior involvement with the Troilus Gold Project that is the subject of the Technical Report.
- My most recent site visit to the Troilus Gold Project described in this report was from July 13th to 15th 2020 for two days.
- As of the date of this Certificate, to my knowledge, information, and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of Troilus Gold Corp. as defined by Section 1.5 of the Instrument.

As of the effective date of the technical report, to the best of my knowledge, information, and belief, the sections of the technical report that I am responsible for, contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Signed and dated this 14th day of October 2020, at Toronto, Ontario.

"signed electronically"

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Gordon Zurowski, P. Eng.

28.2 Andrew Holloway, P.Eng.

I, Andrew Holloway of Nanaimo, Canada, as one of the authors of this technical report titled "Preliminary Economic Assessment of the Troilus Gold Project, Quebec, Canada" dated 14 October 2020, do hereby certify that, and make the following statements:

- I am a Principal Process Engineer with AGP Mining Consultants Inc., with a business address at #246-132K Commerce Park Dr., Barrie ON L4N 0Z7, Canada.
- I graduated from the University of Newcastle upon Tyne, England, B.Eng. (Hons), 1989.
- I am a member in good standing of Professional Engineers of Ontario, membership #100082475.
- I have practiced my profession in the mining industry continuously since graduation.
- My relevant experience with respect to metallurgy and mining project management includes 30 years' experience in the mining sector covering mineral processing, process plant operation, design engineering, and operations and project management.
- I am responsible for Sections 1.7, 1.9, 1.15.4, 13, 17, 21.2.3, 21.3.3, 25.4 and 26.6 of this technical report, titled "Preliminary Economic Assessment of the Troilus Gold Project, Quebec, Canada", and dated 14 October 2020.
- I have had no prior involvement with the Troilus Gold Project that is the subject of the Technical Report.
- No site visit was required.
- As of the date of this Certificate, to my knowledge, information, and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of Troilus Gold Corp. as defined by Section 1.5 of the Instrument.

As of the effective date of the technical report, to the best of my knowledge, information, and belief, the sections of the technical report that I am responsible for, contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Signed and dated this 14th day of October 2020, at Toronto, Ontario.

"signed electronically"

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Andrew Holloway, P. Eng.

28.3 Paul Daigle, géo., P.Geo.

I, Paul Daigle of Toronto, Canada, as one of the authors of this technical report titled “Preliminary Economic Assessment of the Troilus Gold Project, Quebec, Canada” dated 14 October 2020, do hereby certify that, and make the following statements:

- I am an Associate Senior 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 30 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 Boto Gold Project, Senegal and Detour Gold Deposit, Canada.
- I am responsible for Sections 1.2 to 1.6 , 1.15.1, 4 – 12, 14, 23, 25.1, 26.2, 27 of this technical report, titled “Preliminary Economic Assessment of the Troilus Gold Project, Quebec, Canada”, and dated 14 October 2020.
- I have had no prior involvement with the Troilus Gold Project that is the subject of the Technical Report.
- My most recent site visit to the Troilus Gold Project described in this report was from the 18 to 20 of February 2020 for two days.
- As of the date of this Certificate, to my knowledge, information, and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of Troilus Gold Corp. as defined by Section 1.5 of the Instrument.

As of the effective date of the technical report, to the best of my knowledge, information, and belief, the sections of the technical report that I am responsible for, contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Signed and dated this 14th day of October 2020, at Toronto, Ontario.

“electronically signed”

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Paul J. Daigle, géo., P. Geo.