FIRST MINING GOLD

First Mining Gold Corp.

NI 43-101 Technical Report and Pre-Feasibility Study on the Springpole Gold Project, Ontario, Canada

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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
Centipoise	mPa·s
Cubic centimetre	cm^3
Cubic feet per minute	cfm
Cubic feet per second	ft^3/s
Cubic foot	ft^3
Cubic inch	in^3
Cubic metre	m^3
Cubic yard	yd^3
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)	USD$
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 m^2)	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/m^3
Kilograms per hour	kg/h
Kilograms per square metre	kg/m^2
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	Mbm^3
Million metres cubed	Mm^3
Million tonnes	Mt
Minute (plane angle)	'
Minute (time)	min
Month	mo
Ounce	oz
Pascal	Pa
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	cm^2
Square foot	ft^2
Square inch	in^2
Square kilometre	km^2
Square metre	m^2
Thousand tonnes	kt
Three-Dimensional	3D
Tonne (1,000 kg)	t
Tonnes per day	tpd
Tonnes per hour	t/h
Tonnes per year	t/a
Tonnes seconds per hour metre cubed	ts/hm^3
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
Cut-off Grade	COG
Cyanide Soluble	CN
Digital Elevation Model	DEM
Direct Leach	DL
Distributed Control System	DCS
Drilling and Blasting	D&B
Environmental Management System	EMS
Flocculant	floc
Free Carrier	FCA
Gemcom International Inc.	Gemcom
General and Administration	G&A
Gold	Au
Gold Equivalent	AuEq
Heating, Ventilating, and Air Conditioning	HVAC
High Pressure Grinding Rolls	HPGR
Indicator Kriging	IK
Inductively Coupled Plasma 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
Light Detection and Ranging	LiDAR
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
Million Meters Cubed	mm^3
Mineral Deposits Research Unit	MDRU
Material Take Off	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
Non-acid Generating	NAG
Northwest Transmission Line	NTL
Official Community Plans	OCPs
Operator Interface Station	OIS
Ordinary Kriging	OK
Organic Carbon	org
Potassium Amyl Xanthate	PAX
Potentially Acid Generating	PAG
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
Silver	Ag
Standards Council of Canada	SCC

Stanford University Geostatistical Software Library	GSLIB
Terrestrial Ecosystem Mapping	TEM
Total Dissolved Solids	TDS
Total Suspended Solids	TSS
Tunnel Boring Machine	TBM
Underflow	U/F
Valued Ecosystem Components	VECs
Waste Storage Facility	WSF
Water Balance Model	WBM
Work Breakdown Structure	WBS
Workplace Hazardous Materials Information System	WHMIS
X-Ray Fluorescence Spectrometer	XRF

Forward Looking Statements

This Technical Report, including the economics analysis, contains forward-looking statements within the meaning of the United States Private Securities Litigation Reform Act of 1995 and forward-looking information within the meaning of applicable Canadian securities laws. While these forward-looking statements are based on expectations about future events as at the effective date of this Report, the statements are not a guarantee of First Mining's future performance and are subject to risks, uncertainties, assumptions, and other factors, which could cause actual results to differ materially from future results expressed or implied by such forward-looking statements. Such risks, uncertainties, factors, and assumptions include, amongst others but not limited to metal prices, mineral resources, mineral reserves, capital and operating cost forecasts, economic analyses, smelter terms, labour rates, consumable costs, and equipment pricing. There can be no assurance that any forward-looking statements contained in this Report will prove to be accurate, as actual results and future events could differ materially from those anticipated in such statements.

1 SUMMARY

1.1 Introduction

First Mining Gold Corp. (First Mining) is a Canadian exploration and development company, based in Vancouver, Canada, and is publicly-listed on the Toronto Stock Exchange (TSX). First Mining is focused on the development and permitting of the Springpole Gold Project (the Springpole Gold Project or the Project) in northwestern Ontario. First Mining holds a 100% interest in the mineral rights for the Project.

This Technical Report (the Report) and Pre-Feasibility Study (PFS) was prepared for First Mining by AGP Mining Consultants Inc. (AGP), SRK Consulting (Canada) Inc. (SRK), Knight Piésold Ltd. (Knight Piésold), and Swiftwater Consulting Ltd. (Swiftwater) to present the results of the PFS on the Springpole Gold Project. This Technical Report was prepared for the Springpole Gold Project in accordance with NI 43-101 and Form 43-101F1.

The PFS concludes that the Springpole Gold Project could be developed as a phased open pit operation. The pit would feed a 30,000 tpd mill with a total of 121.6 Mt of ore grading 0.97 g/t Au, and 5.23 g/t silver over an 11.3 year mine life. Three years of pre-stripping activity would be required to establish site infrastructure including cofferdams to isolate a small portion of Springpole Lake under which the proposed pit would be located. The process facility would be a conventional crushing and grinding plant followed by a flotation and carbon-in-pulp (CIP) circuit to produce gold doré. Tailings would be filtered and co-disposed with waste rock in a joint facility. 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 USD$1,600/oz, a silver price of USD$20/oz and an exchange rate of 0.75 (CDN$:USD$) the Springpole Gold Project is estimated to have an after-tax IRR of 29.4% and a pay-back period of 2.4 years after the start of production. At a discount rate of 5%, the Project's after-tax NPV is estimated at USD$995M.

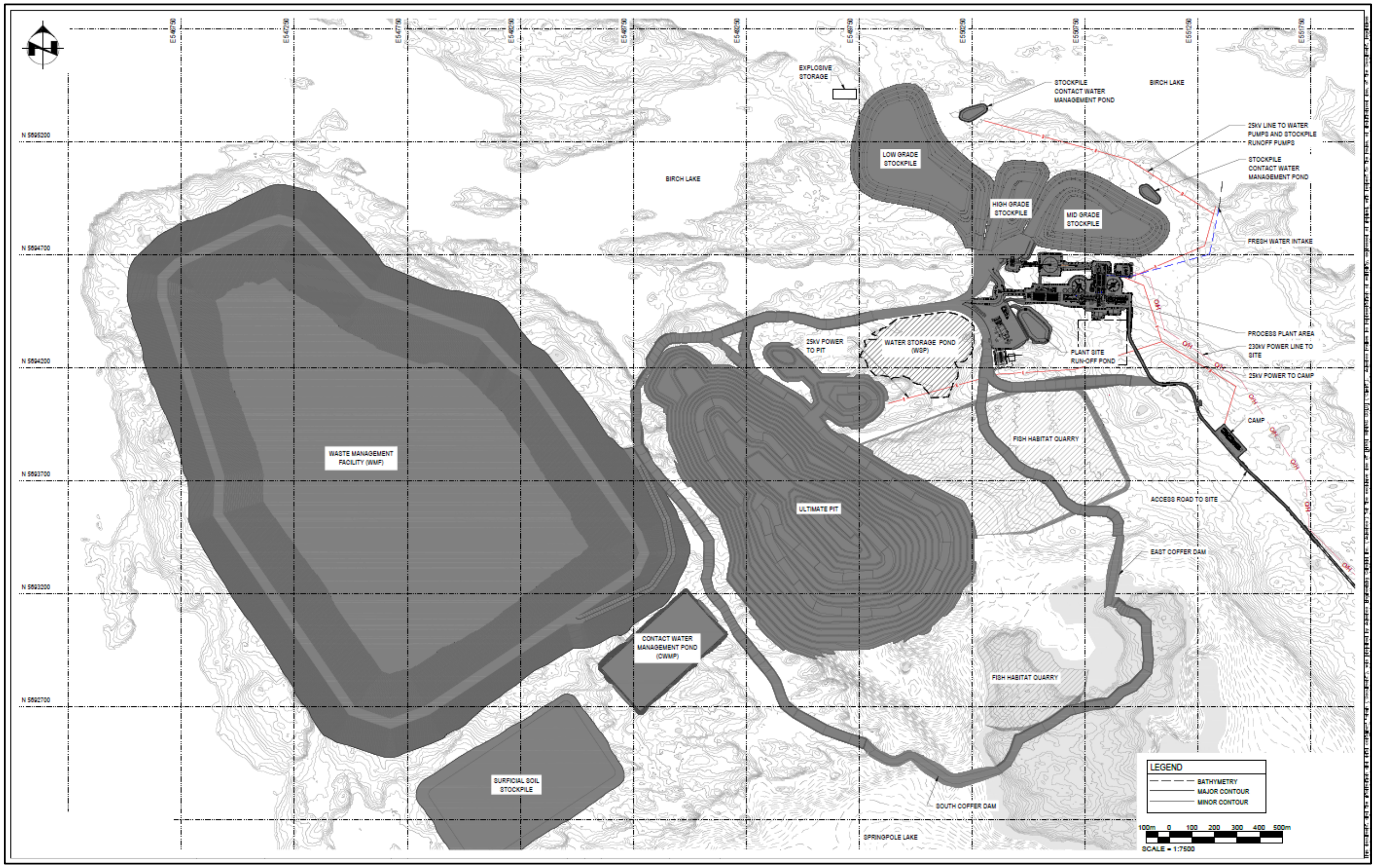
The life of mine capital cost for the Project is estimated at USD$803.3M (CDN$1,071.0M), with an initial capital expenditure of USD$718.3M (CDN$957.7M) which includes USD$79.4M (CDN$105.8M) pre-strip costs which are capitalized.

The PFS utilizes Probable Mineral Reserves for the mine plan which were converted from Indicated Mineral Resources.

Based on the results of the PFS study, AGP recommends that First Mining proceed with a Feasibility Study as part of the development plan for the Springpole Gold Project to help determine a project execution decision. Recommendations and associated budgets are provided in this Technical Report to ensure sufficient information is available going forward.

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

Figure 1-1: Proposed Overall Site Layout



Source: AGP, 2021

1.2 Property Description and Ownership

The Springpole Gold Project is located 110 km northeast of Red Lake, Ontario and is 100% controlled by First Mining. The Project's land position comprises 30 patented claims, 282 contiguous mining claims, and 13 mining leases totalling an area of 41,943 ha.

During late spring, summer, and early fall, the Project is accessible by floatplane direct to Springpole Lake or Birch Lake. During winter, an ice road approximately 85 km long is constructed from the South Bay landing point on Confederation Lake to a point approximately 1 km from the Springpole camp.

1.3 Geology and Mineralization

The Springpole area is underlain by a polyphase alkali, trachyte intrusion displaying autolithic breccia. The intrusion consists of a system of multiple phases of trachyte that is believed to be part of the roof zone of a larger syenite intrusion; fragments displaying phaneritic textures were observed from deeper drill cores in the southeast portion of the Portage zone. Early intrusive phases consist of megacrystic feldspar phenocrysts of albite and orthoclase feldspar in an aphanitic groundmass. Successive phases show progressively finer-grained porphyritic texture while the final intrusive phases are aphanitic. Within the country rocks to the north and east are trachyte and lamprophyre dikes and sills that source from the trachyte- or syenite-porphyry intrusive system.

The main intrusive complex appears to contain many of the characteristics of alkaline, porphyry style mineralization associated with diatreme breccias (e.g. Cripple Creek, Colorado). This style of mineralization is characterized by the Portage zone and portions of the East Extension zone where mineralization is hosted by diatreme breccia in aphanitic trachyte. It is suspected that ductile shearing and brittle faulting have played a significant role in redistributing structurally controlled blocks of the mineralized rock. Diamond drilling in the winter of 2010 revealed a more complex alteration with broader, intense zones of potassic alteration replacing the original rock mass with biotite and pyrite. In the core area of the deposit where fine-grained, disseminated gold mineralization occurs with biotite, the primary potassic alteration mineral, gold, displays a good correlation with potassium/rubidium.

1.4 History

Gold exploration at the Project was carried out during two main periods, the first period was from the 1920s to 1940s, and a second period from 1985 to the present.

Between 1933 and 1936, the Windigokan Sturgeon Mining Syndicate conducted extensive trenching and prospecting, including 10 short holes totalling 458.5 m. The claims were then transferred to Springpole Mines Ltd. who carried out limited trenching and prospecting in 1945.

The area remained dormant until 1985 when Goldfields Canadian Mining Ltd. (GFCM) optioned the Frahm claims and, in 1986, the Milestone claims and the Maple Leaf (now Springpole Group) claims. GFCM conducted an airborne (Aerodat) geophysical survey in 1985 along with geological mapping, humus geochemistry, and ground geophysics.

From 1986 through 1989, GFCM completed 118 diamond drill holes in seven drill phases totalling 38,349 m. In addition, during 1986 and 1987, approximately 116,119 m^2 of mechanical stripping was carried out.

Late in 1989, GFCM entered into a 50/50 joint venture with the combined interests of Noranda and Akiko-Lori Resources Ltd. (Akiko-Lori).

From 1989 through 1992, Noranda conducted an induced polarization (IP) survey over the central portion of the Portage zone and tested the property with eighteen core holes totalling 5,993 m.

During 1992 to 1994, Akiko-Lori/Akiko Gold completed an additional 15 diamond drill holes at the Project totalling 5,154 m.

During 1995 and 1996, Santa Fe drilled an additional 69 core holes totalling 15,085 m on the Springpole Gold Project. After Santa Fe's departure, Gold Canyon Resources Inc. (Gold Canyon) continued exploration at the Springpole Gold Project in 1997 and 1998 with another 52 core holes totalling 5,643 m.

Paso Rico Resources Ltd. (Paso Rico) had an option to earn an interest in the Project and, in 1998 and 1999, conducted a lake bottom sediment sampling program in several areas of Springpole Lake, as well as 12 diamond drill holes totalling 2,779 m with Gold Canyon.

During 2004, 2005, and 2006, diamond drilling programs were conducted at the Project by Gold Canyon. A total of 109 holes were completed during this period, over 17,322 m.

From 2007 to 2013, Gold Canyon completed a further 252 diamond drill holes totalling 89,584 m at the Project.

On November 13, 2015, First Mining (which was called First Mining Finance Corp. at the time) completed the acquisition of Gold Canyon, and as a result, acquired the Springpole Gold Project.

1.5 Exploration

The initial geological and engineering studies at the end of 2009 resulted in the establishment of systematic drill sections at 50 m intervals across the three identified prospect areas, namely the Portage zone, the East Extension zone, and the Camp zone. The subsequently developed drill program led to a multi-phase drill campaign starting in the summer of 2010 and ending in 2013, resulting in the completion of 85,000 m of diamond core drilling across 233 drill holes. During the course of the 2010 - 2013 programs, drilling identified a precious metal deposit of significant strike, depth, and width within the Portage zone.

Between 2016 and 2020, First Mining drilled an additional 18 core holes to test the cofferdam area and to collect additional metallurgical samples.

1.6 Drilling and Sampling

1.6.1 Drilling

During the winters of 2007 and 2008, Gold Canyon conducted diamond drill programs that completed 18 holes totalling 4,574 m.

Between 2010 and 2013, 233 diamond drill holes were drilled for a total of 85,010 m of drilling.

An oriented core drilling program was carried out in 2013 to collect rock geotechnical data within the immediate vicinity of the proposed open pit. Approximately 2,402 m of drilling was completed across seven drill holes. During summer 2013, a further 17 exploration holes were drilled, totalling 2,993 m, and in the fall of 2013, 18 'Vibracore' holes (totalling 721 m) were drilled under Springpole Lake targeting the lake bottom sediments.

The 2016 drill program was implemented by First Mining to collect additional material from the Portage zone so that additional metallurgical testing could be carried out. In total, 1,712 m were drilled across four diamond drill holes.

In 2018, First Mining carried out a limited drill program to test the cofferdam integrity. Eleven short diamond drill holes were completed, totalling 243 m. None of the holes intersected the mineralized domains and none have any impact on the mineral resources at the Project.

Three diamond drill holes totalling 1,182 m were drilled in 2020 to collect additional material for metallurgical testing within the immediate vicinity of the proposed open pit.

1.6.2 Sampling

For Gold Canyon's 2010 and 2011 drill programs, and the 2016 – 2020 First Mining drill programs, all the drill core intervals were sampled using sample intervals of 1 m. During the 2012 drilling program, Gold Canyon changed its standard sample length from 1 to 2 m lengths. However, in zones of poor recovery, 1.5 or 3 m samples were sometimes collected. Samples over the standard sample length were typically half core samples and whole core was generally only taken in intervals of poor core recovery across the sampled interval. Sampling marks were made on the core and sample tickets were stapled into the core boxes at the beginning of each sample interval. Quality control samples were inserted into the sample stream.

All primary assay work since the 2010 drill program has been performed by SGS Laboratories in Red Lake (gold), Ontario, and Don Mills (silver and multi-element) in Toronto, Ontario.

In the opinion of SRK, the sampling preparation, security and analytical procedures used in the drill programs conducted by Gold Canyon for gold analyses are acceptable but not fully consistent with generally accepted industry best practices because of the lack of standard reference material for silver for the earlier drill campaigns. However, because of the relatively low economic value of silver, SRK concludes that the assay data are adequate for use in resource estimation. First Mining has an established quality assurance and quality control (QA/QC) protocol for the acceptance of assay batches with respect to the performance of standard reference material, duplicates, and blanks. SRK also notes that First Mining is now including some standard reference material for silver in their drill programs.

1.7 Data Verification

SRK carried out visits to the Springpole site on February 10 and 11, 2012, and again on August 8 and 9, 2012. During the site visits, core logging procedures were reviewed. Several sections of core from the Portage, Camp, and East Extension zones were examined. Sampling procedures and handling were observed. The deposit geology, alteration, and core recovery data were observed for the Portage zone.

As part of the mineral resource estimation process, SRK reviewed the QA/QC data collected by Gold Canyon, reviewed the procedures in place to assure assay data quality, and verified the assay database against original assay certificates provided directly to SRK by SGS Red Lake, the assay laboratory. A total of 53,431 gold assays, 46% of the assay data, were checked against original assay certificates. No significant database errors were identified. About 143 minor rounding errors were observed. None of the rounding errors are deemed material or of any significance to the mineral resource estimate presented in this report.

1.8 Metallurgical Testwork

During 2020, First Mining completed a comprehensive comminution and metallurgical testwork program to support the PFS. This included head grade analyses, mineralogy, a full suite of comminution tests, flotation and leach tests, cyanide detoxification, rheology, and solid/liquid separation. Testwork was conducted in two phases; Phase 1 used available coarse reject sample from the 2016 drilling campaign and Phase 2 used fresh HQ drill core from the 2020 winter drilling campaign.

Tests were performed on mineralization considered to be representative of material that will be sent to the plant. Composite samples representing major lithologies and a range of head grades aligned with the minimum and maximum values expected in the plant feed in the first 9 years of production.

The grade variability samples gold and silver grades ranged from 0.6 to 2.0 g/t Au and 0.5 to 20.0 g/t Ag.

Bulk mineralogy on select composites showed the main sulphide mineral was pyrite, ranging from 5.3 to 7.7%, with traces of chalcopyrite, sphalerite, and galena. Gold deportment studies indicated 5 - 12% of the gold is sub-microscopic; 8 - 14% of the gold is locked in <11 μm size fraction; 42 - 64% of the gold is exposed and 22 - 32% is liberated. A host of telluride minerals exist in the microscopic size range, with petzite the most dominant. Gold and electrum occur in minor amounts.

Comminution testing showed that the materials tested are considered very soft to medium competency, with semi-autogenous grind (SAG) mill comminution (SMC) tests ranging from 40 to 124 and SAG Power Index (SPI) tests ranging from 7 to 67 min. Conventional Bond tests showed significant variation in hardness, with Bond rod mill work indices ranging from 9 to 15 kWh/t and Bond ball mill work indices ranging from 8 to 18 kWh/t.

Rougher flotation tests showed high sulphide recovery was generally achieved within 8 minutes flotation time. Excessive foaming was observed in some samples. This was attributed to drilling compound added to the core, to aid core recovery. High mass pull was observed in these samples. A cleaning stage reduced the mass pull reporting to concentrate regrind. Flotation recoveries to cleaner concentrate ranged from 55 to 83% for gold, from 55 to 90% for silver, and from 75 to 98% for sulphur at a target mass pull of 15% or less. Leaching of flotation tails is required to attain acceptable gold recovery.

Flotation concentrates gold extraction showed significant benefit at the finer concentrate regrind size of 80% passing 15 to 17 μm. Particularly high residue grades were observed at 80% passing 25 μm. Flotation concentrate gold extractions ranged from 62 to 97%, dependent on feed grade. Flotation tail gold extractions ranged from 52 to 94%.

Overall plant recoveries for gold are predicted to range from 84 to 85% for head grades of 0.8 to 1.22 g/t Au. Overall plant recoveries for silver are predicted to range from 84 to 91% for head grades of 3.2 to 8.3 g/t Ag.

Whole ore cyanide leach tests showed relatively poor extraction at a grind size of 80% passing 75 µm or greater using aggressive telluride leach conditions. Gold leach extractions ranged from 52 to 72%. At a finer grind of 80% passing 60 μm, gold extractions ranged from 64 to 84%.

Cyanide detoxification tests achieved <1 mg/L weakly acid dissociable cyanide (CN_{WAD}), with favourable reagent consumption rates.

Mercury grades were in the range of less than 0.3 to 8 g/t in the flotation feed. A retort with gas collection system was incorporated into the plant design to manage and control mercury in the process. Arsenic is present in the feed at concentrations less than 30 g/t and is not expected to be problematic in processing. No other elements that may cause issues in the process plant or concerns with product marketability were noted.

Thickening and filtration of cyanide detoxified slurry showed a moisture content of 18.5% was achieved with high-rate thickening followed by pressing and drying within a conventional plate and frame filter press. A moisture content of 15% was achieved when employing a membrane squeeze in addition to pressing and drying in a plate and frame filter press.

1.9 Mineral Resource Estimate

The mineral resource model prepared by SRK considers 662 core boreholes drilled by previous owners of the property during the period from 2003 to 2013, and seven holes drilled by First Mining in 2016 and 2020. The resource estimation work was completed by Dr. Gilles Arseneau, Ph.D., P.Geo. (APEGBC #23474) of SRK. The effective date of the mineral resource estimate used in this Report is July 30, 2020.

The revised mineral resource estimate was based on a gold price of USD$1,550/oz and a silver price of USD$20/oz, both considered reasonable economic assumptions by SRK. To establish a reasonable prospect of economic extraction in an open pit context, the resources were defined within an optimized pit shell with pit walls set at 35 to 50° based on domain. Assumed metallurgical recoveries of 88% for gold and 93% for silver were used. Mining costs were estimated at CDN$1.62/t of total material, processing costs estimated at CDN$15.38/t, and general and administrative (G&A) costs estimated at CDN$1.00/t. A cut-off grade (COG) of 0.3 g/t Au was calculated and is considered to be an economically reasonable value corresponding to breakeven mining costs. Approximately 90% of the revenue for the proposed Project is derived from gold, with 10% derived from silver.

Mineral resources were estimated by ordinary kriging using Gemcom block modelling software in 10 m x 10 m x 6 m blocks. Grade estimates were based on capped, 3 m composited assay data. Capping levels were set at 25 g/t for Au and 200 g/t for Ag. Blocks were classified as Indicated Mineral Resources if at least two drill holes and six composites were found within a 60 m x 60 m x 40 m search ellipse. All other interpolated blocks were classified as Inferred Mineral Resource. Mineral resources were then validated using Gemcom GEMS (6.7) software.

The resource model includes mineralized material in the Main, East Extension and Portage zones spanning approximately 1,860 m in the southeast direction along the axis of the Portage zone and 900 m in the northeast direction across the Portage zone. The resource modelling includes mineralized material generally ranging from the surface to a depth of 340 - 440 m below surface.

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, taxation, sociopolitical, marketing, or other relevant issues. There has been insufficient exploration to define these Inferred Mineral Resources as an Indicated or Measured Mineral Resource but SRK is of the opinion that with additional drilling that the majority of the Inferred Mineral Resources could be upgraded to Indicated Mineral Resources.

The mineral resources in this Report were estimated in accordance with current Canadian Institute of Mining, Metallurgy and Petroleum (CIM) standards definitions, and guidelines, and reported using the 2014 CIM Definition Standards. The updated mineral resource estimate for the Springpole Gold Project is summarized in Table 1-1.

Table 1-1: Mineral Resource Statement, Inclusive of Mineral Reserves, Springpole Gold Project (July 30, 2020)

Category	Quantity	Grade		Metal	
	(Mt)	Au (g/t)	Ag (g/t)	Au (Moz)	Ag (Moz)
Open Pit**					
Indicated	151	0.94	5.0	4.6	24.3
Inferred	16	0.54	2.8	0.3	1.4

*Mineral resources are reported in relation to a conceptual pit shell. Mineral resources that are not mineral reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate. All composites have been capped where appropriate.

*Open pit mineral resources are reported at a COG of 0.3 g/t Au. COGs are based on a gold price of USD$1,550/oz and a gold processing recovery of 88% and a silver price of USD$20/oz and a silver processing recovery of 93%.

*Mining costs were estimated at CDN$1.62/t of total material, processing costs estimated at CDN$15.38/t, and general and administrative (G&A) costs estimated at CDN$1.00/t.

*Pit slope angles ranged from 35 - 50°.

1.10 Mineral Reserve Estimate

The mineral reserve estimate for Springpole is based on the conversion of the Measured and Indicated Mineral Resources within the current Technical Report mine plan. Indicated Mineral Resources in the mine plan were converted directly to Probable Mineral Reserves. There are currently no Measured Resource estimates for the Project and therefore there are no Proven Mineral Reserves. The total mineral reserves for the Springpole Gold Project are shown in Table 1-2.

Table 1-2: Springpole Mineral Reserves – Proven and Probable

Category	Tonnes (Mt)	Grade		Contained Ounces	
		Au (g/t)	Ag (g/t)	Au (Moz)	Ag (Moz)
Proven	0.0	0.00	0.00	0.00	0.0
Probable	121.6	0.97	5.23	3.80	20.5
Total	**121.6**	**0.97**	**5.23**	**3.80**	**20.5**

* This Mineral Reserve estimate has an effective date of December 30, 2020 and is based on the Springpole Gold Project Mineral Resource estimate that has an effective date of July 30, 2020. The Mineral Reserve estimate was completed under the supervision of Gordon Zurowski, P.Eng., of AGP, who is a Qualified Person (QP) as defined under NI 43-101. Mineral Reserves are stated within the final design pit based on a USD$878/oz Au price pit shell with a USD$1,35 /oz Au price for revenue. The equivalent COG was 0.34 g/t Au for all pit phases. The mining cost averaged CDN$2.75/t mined, processing costs averaged CDN$14.50/t milled, and G&A was CDN$1.06/t milled. The process recovery for gold averaged 88% and the silver recovery was 93%. The exchange rate assumption applied was CDN$1.30 equal to USD$1.00.

*Pit slope angels ranged from 35 - 50°.

The mineral reserves for the Springpole Gold Project are based solely on open pit mining assumptions.

The QP has not identified any known legal, political, environmental, or other risks that would materially affect the potential development of the mineral reserves at the Springpole Gold Project. The risk of not being able to secure the necessary permits from the government for development and operation of the Project exists, but the QP is not aware of any issues that would prevent those permits from being withheld per the normal permitting process.

1.11 Mining Methods

The PFS is based on open pit mining of the proposed Springpole pit. This pit will provide feed material necessary to maintain the process plant feed rate at 30,000 tpd while operational.

The Springpole pit will be a three phased pit which will provide 121.6 Mt of ore grading 0.97 g/t Au, and 5.23 g/t Ag. Waste from this pit will total 275.4 Mt for a strip ratio of 2.3:1 (waste:ore). With the inclusion of the proposed quarry, the total waste movement will be 287.5 Mt for a life-of-mine (LOM) strip ratio of 2.36:1 (waste:ore)

In addition to the pit, a quarry would be established near the plant location in the pre-production period. This quarry would be used to construct mine infrastructure including haul roads, cofferdams and to meet site fill requirements for other infrastructure.

The mill feed cut-off used is 0.40 g/t Au. During the mine operation material would be stockpiled to optimize the plant feed grade and defer lower-grade material until later in the mine schedule. The three grade bins used for the stockpiles included: low grade (0.40 to 0.60 g/t Au), medium grade (0.60 to 0.80 g/t Au) and high grade (over 0.80 g/t Au).

The phases are scheduled to provide 30,000 tpd of feed to the mill over an 11.3 year mine life after three years of pre-production stripping. The first two years of pre-production stripping are construction related. The last three years of mining are stockpile reclaim. The pits are sequenced to minimize initial stripping and provide higher feed grades in the early years of the mine life which the stockpiling strategy accomplishes.

The pits will be built on 12 m benches with safety berms placed every 24 m. Inter-ramp angles vary from 39 to 54° in rock depending upon the wall orientation. Overburden will use a 30° inter-ramp angle with 12 m between berms. Minimum mining widths of 35 to 40 m were maintained in the design with preferred bench widths of 60 m or more. Ramps will be at maximum 10% gradient and vary in width from 27.1 m (single lane width) to 35.4 m (double lane width). They have been designed for a 226 t haulage truck.

The main fleet will consist of three 251 mm rotary drills, two 36 m^3 electric hydraulic shovels and one 23 m^3 front-end loader. The truck fleet will total seventeen 240 t trucks at the peak of mining. This is due to the long hauls from the pit to the waste storage facilities (WSF) as well as the backhaul of tailings material from the plant. 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.

In the pre-production years -3 and -2, 3.9 Mt will be mined within the quarry area. This mining will be with 91 t trucks, 6 m^3 excavators and smaller track drills, more suited to this type of work, preparing the site for the larger, more productive, equipment. Year -1 is the start of major mining activity using the larger equipment when the bay dewatering has advanced sufficiently for mining and the site infrastructure (power lines, roads, etc.) is in place. The early phases provide the highest grade to the mill early in the schedule. The open pit will be in operation until Year 9 followed by three years of stockpile reclaim to feed the plant. When the open pit is complete, the larger mining fleet will move to complete the quarry area, dumping the material into the open pit. This will serve to cover the slopes in the pit for reclamation purposes.

Waste material from the pit will be stored in the WSF. Non-acid generating (NAG) material will be used for the outer berms while potentially acid generating (PAG) material will be co-mingled with filtered tails. The filtered tails will be backhauled from trucks returning from dropping material at the plant either as feed or placed in the stockpiles. As the WSF advances upwards, re-sloping of the sides will be occurring to allow for concurrent reclamation and reducing the visual impact of the facility. The majority of the waste rock will be contained within the WSF (196.6 Mm^3), but a small portion of NAG material will be backfilled into Phase 2 of the open pit near the end of the mine life. This will reduce the overall haul length and will help in pit reclamation. A total of 9.8 Mm^3 will be backfilled into the pit.

1.12 Recovery Methods

The process plant was designed using conventional processing unit operations. It will treat 30,000 tpd or 1,250 t/h based on an availability of 8,059 hours per annum or 92%. The crusher plant section design is set at 75% availability and the gold room availability is set at 52 weeks per year including two operating days and one smelting day per week. The plant will operate with two shifts per day, 365 days per year, and will produce gold doré bars.

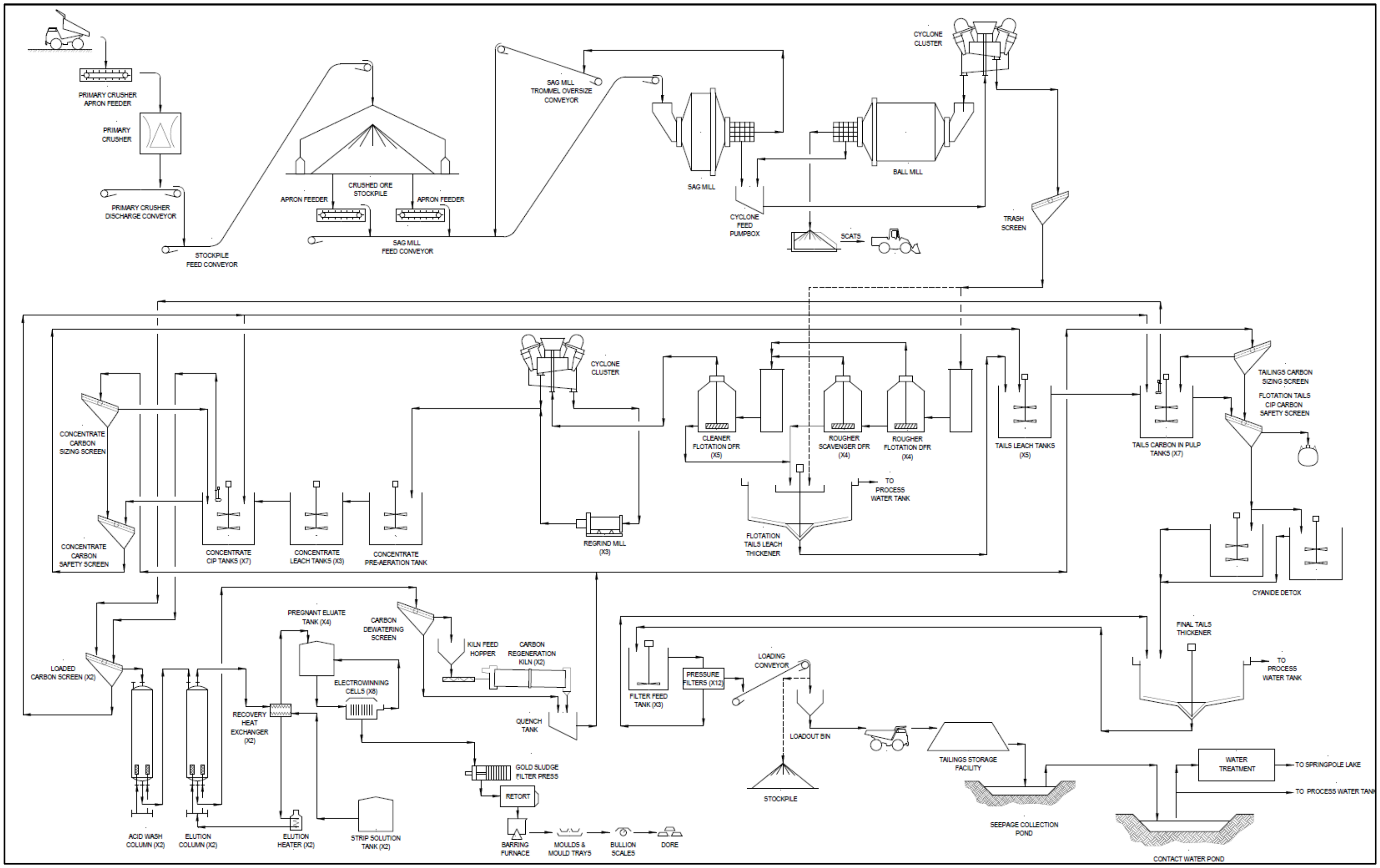
The plant feed will be hauled from the mine to a crushing facility that will include a gyratory crusher as the primary stage before being conveyed to the crushed ore stockpile. The crushed ore will be ground by a SAG mill, followed by a closed circuit of a ball mill with a hydro-cyclone cluster. The hydro-cyclone overflow with P_{80} of 150 mesh (106 µm) will flow to a three-stage flotation circuit including rougher flotation, rougher scavenger flotation, and cleaner flotation. Flotation tailings will report to the tailings leaching and CIP circuit. Flotation concentrate will report to a closed loop cyclone cluster and IsaMill before reporting to the concentrate leach and CIP circuit.

Gold and silver leached in the CIP circuits will be recovered onto activated carbon and eluted in a pressurized Anglo American Research Laboratory (AARL)-style elution circuit and then recovered by electrowinning in the gold room. The gold –silver precipitate will be dried in a mercury retort oven and then mixed with fluxes and smelted in a furnace to pour doré bars. Carbon will be re-activated in a carbon regeneration kiln before being returned to the CIP circuits. CIP tails will be treated for cyanide destruction prior to pumping to a final tails thickener and pressure filter. Filter cakes will be hauled to the waste storage facility (WSF) for disposal.

The installed power for the process plant will be 58 MW and the power consumption is estimated to be 32 kWh/t processed. Raw water will be pumped from Birch Lake to a raw-water storage tank. Potable water will be sourced from the raw-water tank and treated in a potable water treatment plant. Gland water will be supplied from the raw-water tank. Process water will primarily consist of water reclaimed from the final tails thickener and pressure filters. Reagents will include pebble lime, sodium cyanide, sodium hydroxide, copper sulphate pentahydrate, hydrochloric acid, sodium metabisulphite, activated carbon, flocculant, coagulant, collector (PAX), and frother (MIBC).

The selected flowsheet is shown in Figure 1-2.

Figure 1-2: Process Flowsheet – Springpole Mine



Source: SRK, 2021

1.13 Project Infrastructure

1.13.1 Infrastructure

Key project infrastructure as envisaged in the PFS includes: open pit mine area including mine haul roads and ramps; cofferdams for hydraulic isolation of the mine pit following bay dewatering; site main access roads, administrative access roads and maintenance roads, site main gate and guard house; administration and dry building, construction and permanent camp accommodations; process plant e-room; crushing area e-room; control room; reagent storage building; gold room; assay laboratory and sample preparation area; plant workshop and warehouse; truck shop and warehouse, tire changing facility, truck wash building; fuel facility, fuel storage and dispensing; fresh water intake; 230 kV overland and 25 kV underground power distribution lines; fresh water intake pumping supply and treatment; waste storage facility (WSF), contact water collection ponds; waste water treatment plant and explosives magazine.

The main access road will be a private extension of the existing Wenesaga Road for forestry services and has been constructed up to 15 kilometres from the Project site.

Approximately 58 MW of electrical demand will be supplied via a new 230 kV overhead transmission line, built to connect to the provincial grid's 230 kV line approximately 75 km to the southeast. A 230kV / 25kV transformer will provide step down prior to feeding a total of six electrical rooms. Variable frequency drives have been allowed where required and all medium-voltage motors or drives will be supplied in 4.16 kV.

1.13.2 Cofferdams

Two cofferdams will be constructed to isolate the area of the proposed open pit and facilitate mining following dewatering. The cofferdams will be constructed using NAG rockfill from pre-production phase open pit stripping (sourced from an area outside the existing lake), with a width of 28 m and combined length of 940 m. The maximum constructed cofferdam height will be approximately 17 m. A secant pile wall (SPW) and grout curtain will be installed within the rockfill to establish a hydraulic barrier.

1.13.3 Waste Storage Facility

A single WSF will be constructed west of the open pit for storage of tailings produced from mineral processing and PAG waste rock generated from open pit mining. The WSF will store approximately 76 Mm^3 of tailings and 41 Mm^3 of PAG waste rock within a cell. Structural stability of the facility will be provided by perimeter embankment dams constructed with NAG waste rock generated from open pit mining. The perimeter embankments will be constructed with downstream raises to a maximum height of approximately 70 m. Surface water run-off from the facility will be removed and stored in a contact water management pond (CWMP), to be located south of the WSF, to limit infiltration of water into the waste materials following placement. An engineered cover is conservatively considered in closure, to promote surface run-off and limit seepage, and will be further evaluated through the Environmental Assessment (EA) process.

1.14 Environmental Permitting and Social Considerations

1.14.1 Environmental and Social Setting

First Mining, and its predecessor Gold Canyon, have been actively collecting environmental baseline data necessary to support the Project's EA since 2010.

The Project site is in a remote area of northwestern Ontario. There are no adjoining anthropogenic sources of industrial air emissions. Temperatures at the Project site range between a minimum of -40°C in January to a maximum of 40°C in July. The property is overlain by glaciated terrain common throughout the Canadian Shield. Land areas are generally of low relief with less than 30 m of local elevation change. Tree cover consists of mature spruce, balsam, birch, and poplar. Black spruce and muskeg swamps occupy low-lying areas. Glacial till is generally thin across the site (less than 4 m in thickness). Outcrops are limited and small and are generally covered by a thick layer of moss or muskeg. Land areas are separated by a series of interconnected shallow ponds and lakes. There is generally low to moderate relief in the vicinity of the Project, with generally dry uplands and poorly drained lowland valleys with thick accumulations of organic soils. Experience in this environment suggests that groundwater flow will mostly follow surface water flow directions. From the monitoring data it was observed that the highest flows were measured in the spring following freshet, and the lowest flows in the late summer and fall before freeze-up.

Springpole Lake, as well as the nearby Birch and Seagrave Lakes support diverse fish communities including sport species common to both cold-water lakes (lake trout and whitefish) and cool-water lakes (walleye, northern pike, and yellow perch), and several of other non-game and forage fish. Among the 12 small lakes surveyed; six host fish communities that include sport species including yellow perch and northern pike; four host only forage species; and two are considered devoid of fish.

Forest composition at the Project is typical of the Lac Seul Upland. Dominant tree species include trembling aspen, black spruce, white birch, balsam fir, and white spruce and jack pine. Understory ground cover species composition and abundance is typical of mesic mixed wood boreal sites and lacks microhabitats likely to harbor rare vascular plant species.

The Project area is within the Churchill Caribou Range, and numerous species are present within the region, including seven local wildlife species such as woodland caribou, moose, marten, lynx, snowshoe hare, fisher, and northern flying squirrel, as well as many species of songbirds.

The Red Lake area has been a historic mining district since the gold rush of the 1920s, and it currently has five (5) active mining projects and other decommissioned/abandoned mines situated within the Municipality. The Project is within the Trout Lake Forest SFL and forestry activities are ongoing in the region, in accordance with the Crown Forest Sustainability Act.

The Project is located approximately 40 km from Cat Lake First Nation, 45 km from Slate Falls First Nation, and 120 km from the community of Hudson in the Lac Seul First Nation. First Mining is in consultation with eight Indigenous communities (including seven First Nations and the Metis Nation of Ontario) and will continue this consultation during the EA process.

1.14.2 Environmental and Social Impacts

The area of Springpole Lake that will be dewatered spans approximately 150 hectares and displays significant variation in lakebed elevation, with the deepest point reaching an approximate maximum depth of 40 m (El. 353 masl). This activity will affect fish habitat. First Mining will continue working with Fisheries and Oceans Canada (DFO) to develop off-setting measures that will help to mitigate any short or long-term effects to local fish communities.

First Mining will fully consider the concerns and issues associated with potential adverse environmental effects, as appropriate, to the Indigenous Peoples in terms of proximity, historic resources, land and resource use, physical and social effects (including health) on their communities, as well as economy, employment, cultural heritage, in the EA process.

Preliminary environmental design criteria have been developed for Project features that have the potential to release contaminants into the air, water, and land. First Mining will also develop an environmental, health and safety ("EHS") management system to address the EHS needs of the Project based on the results of the Environmental Impact Statement.

1.14.3 Waste and Water Management

Approximately 76 Mm^3 of tailings and 41 Mm^3 of PAG waste rock will be stored within the Waste Storage Facility (WSF). The construction of perimeter embankment dams using NAG waste rock, will improve stability and provide freeboard for run-off collection as well as storage for storm events.

The surface of the filtered tailings and PAG waste rock within the WSF will be graded to encourage flow to defined sumps/pumping points. The collected surface water will be directed to a Contact Water Management Pond (CWMP) located southeast of the WSF. The water stored in the CWMP will be used to supplement mineral processing and/or treated and released to Springpole Lake.

1.14.4 Environmental Assessment Process

On February 23, 2018, First Mining submitted a Project Description to the Impact Assessment Agency of Canada (IAAC). IAAC determined an EA is required for the Springpole Gold Project under the *Canadian Environmental Assessment Act* (2012). First Mining has also entered into a Voluntary Agreement with the Ontario Ministry of Environment, Conservation and Parks (MECP) to undertake an Individual EA under Section 3.0.1 of the provincial *Environmental Assessment Act*.

First Mining plans to submit an Environmental Impact Statement (EIS) for the Project by the end of 2021. The EIS would be developed to also meet the regulatory requirements associated with the provincial Voluntary Agreement to undertake an individual EA.

1.14.5 Engagement and Consultation

The federal government identified Cat Lake First Nation, Slate Falls First Nation, Lac Seul First Nation, Wabauskang First Nation, Mishkeegogoamang Ojibway Nation, Ojibway Nation of Saugeen, and Métis Nation of Ontario in 2018 (updated in 2020), while in 2018 the provincial government identified Cat Lake First Nation, Slate Falls First Nation, Lac Seul First Nation, Wabauskang First Nation, Mishkeegogoamang Ojibway Nation, Ojibway Nation of Saugeen, Pikangikum First Nation, and Métis Nation of Ontario, as potentially impacted by the Project or having an interest in the Project.

In March 2017, the First Nations of Cat Lake, Slate Falls and Lac Seul entered into a Shared Territory Protocol Agreement. These three First Nations are known collectively as the Shared Territory Protocol Nations ("STPN"). In February 2018, First Mining entered into a Negotiation Protocol Agreement with the STPN and will continue information sharing and consultation throughout the EA process.

1.14.6 Mine Closure and Rehabilitation

The Closure Plan for the Springpole Gold Project had not been written when this Report was finalized. The general rehabilitation and closure approach for the Project will meet the objectives of the Ontario Mines Act and Regulation 240/00. The overall objective for closure is to return the Project site to a productive condition after mining is complete that is capable of supporting plant, wildlife and fish communities, and other applicable land uses. This cost for closure has been estimated at CDN$39 million.

1.15 Markets and Contracts

The PFS used the following metal prices for the base case economic analysis:

- Gold USD$1,600/oz
- Silver USD$20.00/oz

The gold markets are mature global markets with reputable smelters and refiners located throughout the world. First Mining has not completed any formal marketing studies with regard to gold production that will result from the mining and processing of gold ore from the Springpole Gold Project into doré bars. Gold production is expected to be sold on the spot market. Terms and conditions included as part of the sales contracts are expected to be typical of similar contracts for the sale of doré throughout the world.

Doré will be shipped from site to major refineries. First Mining will enter into refining agreements with various refiners around the world when the timing is appropriate. The terms and conditions will be consistent with standard industry practices.

Refining charges include treatment and transportation.

1.16 Capital Cost Estimates

The capital cost estimate has an accuracy of -20% / +30% (AACE Class 4). The estimate includes the cost to complete the design, procurement, construction, and commissioning of all the identified facilities. The estimate was based on the traditional engineering, procurement, and construction management (EPCM) approach where the EPCM contractor would oversee the delivery of the completed project from detailed engineering and procurement to handover of a working facility.

The estimate was derived from a several fundamental assumptions as indicated in process flow diagrams, general arrangements, mechanical equipment list, electrical equipment list, material take offs (MTOs), electrical layouts, scope definition and a work breakdown structure. The estimate included all associated infrastructure as defined by the scope of work.

The capital cost estimate for the Springpole Gold Project is summarized in Table 1-3.

Table 1-3: Springpole Capital Cost Estimate (USD$)

Cost Type	Cost Description	Project Capital (USD$ M)		
		Initial	Sustaining	Total
Direct	Mine	144.5	51.3	195.8
	Site Development	21.0	-	21.0
	Process Plant	296.7	4.2	300.9
	On-site Infrastructure	38.4	-	38.4
	Off-site Infrastructure	35.3	-	35.3
	Direct Subtotal	**535.9**	**55.5**	**591.4**
Indirect	Indirects	47.9	-	47.9
	EPCM Services	37.5	-	37.5
	Owner's Costs	16.1	-	16.1
	Indirect Subtotal	**101.4**	**-**	**101.4**
Provisional	Contingency and Management Reserve	80.9	-	80.9
Closure	Closure Costs	-	29.5	29.5
Total		**718.3**	**85.0**	**803.3**

1.17 Operating Cost Estimates

The operating costs for a mine at the Project have been estimated from base principles with vendor quotations for repair and maintenance costs and other suppliers for consumables. Key inputs to the mine cost are fuel and labour. The price provided for the Project was CDN$0.80/L (USD$0.60/L) delivered to the site. The mine truck and support equipment fleets will be diesel powered. The large production drills, hydraulic shovels and dewatering pumps will be electric powered, and the cost estimate used an electricity price of CDN$0.08/kWh (USD$0.06/kWh).

Labour costs are based on an Owner-Operated scenario with First Mining responsible for the maintenance of the equipment with its own employees.

The mining fleet will be leased to help lower capital costs and payments are included in the operating cost. The mining cost is shown as both cost per tonne mined and cost per tonne moved. This is due to the large quantity of tailings backhaul included in the operating cost. The cost per tonne mined is CDN$2.75/t mined (USD$2.06/t mined) or CDN$1.94/t moved (USD$1.46/t moved). The cost per tonne milled over the LOM is CDN$8.69/t milled (USD$6.52/t milled).

The annual process operating cost is estimated at CDN$158.8 M (USD$119.1 M) and will average CDN$14.50/t milled (USD$10.87/t milled) over the LOM.

The general and administrative (G&A) cost is estimated at CDN$11.57 M (USD$8.68 M) and will average CDN$1.06/t milled (USD$0.79/t milled) over the LOM.

The life of mine operating cost estimate for the Springpole Gold Project is shown in Table 1-4.

Table 1-4: Operating Cost Estimate Summary (USD$)

Operating Cost	Life of Mine Cost (USD$ M)	Cost (USD$/t Processed)
Mining	793	6.52
Processing	1,323	10.87
G&A	96	0.79
TOTAL	**2,212**	**18.18**

1.18 Economic Analysis

The mine plan is based on Indicated Mineral Resources that have been converted to Probable Mineral Reserves.

An economic model was developed to estimate annual pre-tax and post-tax cash flows and sensitivities of the Project based on a 5% discount rate. It must be noted that tax estimates involve complex variables that can only be accurately calculated during operations and, as such, the after-tax results are approximations. A sensitivity analysis was performed to assess the impact of variations in metal prices, head grades, initial capital cost, total operating cost, foreign exchange rate, and discount rate.

The capital and operating cost estimates developed specifically for this Project are in Canadian dollars (CDN) and converted with the stated exchange rate. The economic analysis has been run on a constant dollar basis with no inflation.

The economic analysis was performed using the following assumptions:

- gold price of USD$1,600/oz, silver price of USD$20/oz
- mine life (LOM) of 11.3 years
- exchange rate of USD$0.75 per CDN$1.00 was assumed
- cost estimates in constant Canadian dollars with no inflation or escalation
- 100% ownership with 1.3% net smelter returns (NSR) royalty; (assumes buy back of 1.4% NSR)
- capital costs funded with 100% equity (no financing costs assumed)
- closure cost of USD$29 M
- Canadian corporate income tax system consists of 15% federal income tax and 10% provincial income tax
- Ontario applies a mining tax rate of 10%
- total undiscounted tax payments are estimated to be USD$720 M over the LOM

The pre-tax net present value (NPV) discounted at 5% is USD$1,482 M; the internal rate of return (IRR) is 36.4%; and payback period is 2.2 years. On an after-tax basis, the NPV discounted at 5% is USD$995 M; the IRR is 29.4%; and the payback period is 2.4 years.

A summary of the Project economics is provided in Table 1-5.

Table 1-5: Springpole Gold Project – Discounted Cashflow Financial Summary

General	Units	LOM Total / Avg.
Gold Price	**USD$/oz**	**1,600**
Silver Price	**USD$/oz**	**20.00**
FX	**CDN$:USD$**	**0.75**
Production		
Mine Life	yr.	11.3
Mined Ore	kt	121,636
Mined Waste	kt	287,532
Strip Ratio	w:o	2.36
Daily Throughput	tpd	30,000
Total Mill Feed	kt	121,636
Gold		
Mill Head Grade Au	g/t	0.97
Mill Recovery Au	%	85.7%
Total Payable Ounces Au	koz	3,225
Average Annual Payable Au	koz	287
Silver		
Mill Head Grade Ag	g/t	5.2
Mill Recovery Ag	%	89.5
Total Payable Ounces Ag	koz	18,117
Average Annual Payable Ag	koz	1,610
Operating Cost		
Mining – mined	USD$/t mined	2.06
Mining - milled	USD$/t milled	6.52
Processing	USD$/t milled	10.87
G&A	USD$/t milled	0.79
Total	USD$/t milled	18.18
Capital Cost		
Initial Capex	USD$M	718
Sustaining Capex	USD$M	55
Closure Cost	USD$M	29
Operating Costs per Ounce		
Cash Costs (net)	USD$/oz	618
AISC (net)	USD$/oz	645
Cash Costs	USD$/oz AuEq	673
AISC	USD$/oz AuEq	698
Pre-Tax Economics		
NPV (5%)	USD$M	1,482
IRR	%	36.4
Post-Tax Economics		
NPV (5%)	USD$M	995
IRR	%	29.4
Payback	yr.	2.4

* Cash costs consist of mining costs, processing costs, mine-level G&A and refining charges and royalties.
* All-In Sustaining Costs (AISC) includes cash costs plus sustaining capital and closure costs. AISC is at a project-level and does not include an estimate of corporate G&A.

1.18.1 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and after-tax NPV and IRR, using the following variables: gold price, mill head grades, initial capital cost, operating cost, metallurgical recovery, and discount rate. The analysis showed that the Project is most sensitive to, in order from most to least sensitive: mill head grade; gold price; mining cost; processing cost and initial capital costs. A summary of sensitivity analysis is provided in Table 1-6.

Table 1-6: Sensitivity Summary

Post-Tax NPV (USD$M)						
	Base	**(30.0%)**	**(15.0%)**	**--**	**15.0%**	**30.0%**
Mining Cost	995	1,109	1,052	995	938	880
Processing Cost	995	1,175	1,086	995	902	809
Initial Capex	995	1,277	1,151	995	810	596
Gold Price	995	260	629	995	1,357	1,718
Head Grade	995	216	607	995	1,378	1,759
Post-Tax IRR						
	Base	**(30.0%)**	**(15.0%)**	**--**	**15.0%**	**30.0%**
Mining Cost	29.4%	31.6%	30.6%	29.4%	28.3%	27.2%
Processing Cost	29.4%	32.4%	31.0%	29.4%	27.9%	26.2%
Initial Capex	29.4%	55.9%	40.4%	29.4%	21.4%	15.1%
Gold Price	29.4%	13.1%	22.0%	29.4%	36.1%	42.0%
Head Grade	29.4%	11.9%	21.6%	29.4%	36.4%	42.5%

Pre-Tax NPV (USD$M)						
	Base	**(30.0%)**	**(15.0%)**	**--**	**15.0%**	**30.0%**
Mining Cost	1,482	1,109	1,052	995	938	880
Processing Cost	1,482	1,175	1,086	995	902	809
Initial Capex	1,482	1,277	1,151	995	810	596
Gold Price	1,482	260	629	995	1,357	1,718
Head Grade	1,482	216	607	995	1,378	1,759
Pre-Tax IRR						
	Base	**(30.0%)**	**(15.0%)**	**--**	**15.0%**	**30.0%**
Mining Cost	36.4%	31.6%	30.6%	29.4%	28.3%	27.2%
Processing Cost	36.4%	32.4%	31.0%	29.4%	27.9%	26.2%
Initial Capex	36.4%	55.9%	40.4%	29.4%	21.4%	15.1%
Gold Price	36.4%	13.1%	22.0%	29.4%	36.1%	42.0%
Head Grade	36.4%	11.9%	21.6%	29.4%	36.4%	42.5%

1.19 Risks and Opportunities

The Springpole Gold Project has various risks and opportunities with its current level of information. These are shown in Table 1-7 and Table 1-8.

Table 1-7: Springpole Gold Project – Risks

Project Risk Area	Description	Possible Outcome	Mitigation
Mining	SW Pit Wall Slope	Flatter slopes may be required due to poor rock quality	Additional geotechnical drilling to accurately estimate expected slopes
	PAG/NAG Characterization	Increased PAG material could result in a larger WSF height and reduced NAG material for berm construction	Additional geochemical testwork to improve material understanding
	Water Inflows	Additional pumping requirements and site discharge to environment	Additional hydrogeological drilling/modelling to better understand water inflows
Mineral Processing	Tailings Filtration	Increased capital in filtration plant to generate filtered tailings that can be handled with haul trucks	Additional filter test work to properly size the filter plant for the various plant feed types
Waste Storage Facility	Foundation conditions are worse than expected	Increased overburden removal costs, material not suitable for optional liner placement resulting in higher costs	Detailed site investigation of WSF site to properly categorize
Cofferdam	Foundation conditions worse than expected	May require additional cost to prepare foundation or ensure leakage is not above design	Additional site investigations

Table 1-8: Springpole Gold Project – Opportunities

Project Opportunity Area	Description	Possible Outcome	Achieved
Geology	Inferred material	Conversion of current waste material to Indicated and possible mill feed	Infill drilling to assist in resource classification
Mining	SW Pit Wall Slope	Steeper slopes which will help reduce waste movement	Additional geotechnical drilling to accurately estimate expected slopes
	PAG/NAG Characterization	Reduced PAG material could result in a smaller WSF height	Additional geochemical testwork to improve material understanding
	PAG material stored in Pit backfill	Reduced size of WSF Potential improved mine closure situation	Geochemical testwork to mimic subaqueous disposal for permitting
Mineral Processing	Tailings Filtration	Decreased plant capital in filtration	Additional filter test work to properly size the filter plant for the various plant feed types
Waste Storage Facility	Foundation conditions are better than expected	Reduced overburden removal costs, reduced quarry material requirements	Detailed site investigation of WSF site to properly categorize
Cofferdam	Foundation conditions better than expected	Reduced time for construction improving overall project schedule	Additional site investigations

1.20 Recommendations

The PFS for the Springpole Gold Project has indicated a positive Project. AGP recommends that First Mining proceed forward with additional studies for the Project including a Feasibility Study (FS). The recommendations and associated budgets by area are described further in the following sub-sections.

A summary of the expected study costs is shown in Table 1-9.

Table 1-9: Recommended Study Budget

Area of Study	Approximate Cost (CDN$)
Geology	922,000
Geotechnical	400,000
Metallurgy	1,200,000
Infrastructure	2,000,000
Environmental	800,000
Feasibility Study	3,000,000
TOTAL	**8,322,000**

1.21 Geology

1.21.1 Inferred Resource Upgrade

In various pit areas it was noted that with a short drill program, some of the material currently classed as Inferred Mineral Resources, could potentially be converted to Indicated Mineral Resources. This could result in material currently destined for the WSF being processed. The bulk of the Inferred Mineral Resources at the Project are in the southern part of the pit, and approximately 4,100 m of drilling over 16 holes would be required to potentially support the upgrade of the material in this area to the Indicated Mineral Resource category.

1.21.2 Density Measurements

As part of the PFS work, an additional 471 core samples underwent bulk density testing. While sufficient for a PFS study, additional bulk density testwork is recommended to further improve the density model. This could be completed on existing core kept in storage on site at the Project.

1.21.3 Geology Estimated Budget

The following is the estimated budget for the proposed drilling program and other study work for the continued development of the Mineral Resource estimates at the Springpole Gold Project. The estimated budget for these proposed programs would be CDN$922,000 as shown in Table 1-10.

Table 1-10: Estimated Budget of Proposed Work

Proposed Work		Approximate Cost (CDN$)
Inferred Material Upgrade		
Diamond Drilling (~4,100 m)	$200/m	820,000
Bulk Density Measurement; 500 samples	$35/sample	17,500
Subtotal		837,500
Contingency		84,500
TOTAL		**922,000**

1.22 Geotechnical

The results of the geotechnical gap analysis indicate a number of important factors that require additional investigation. For Feasibility-level designs, a higher level of confidence is required, and the preliminary geotechnical model presented will need to be updated with additional data and perhaps more importantly, additional integrated structural geotechnical interpretation and analyses. Recommended data collection and interpretation tasks are outlined in the following sub-sections. These recommendations could potentially be completed in phases, using combinations of third-party consultants and First Mining staff.

The following geotechnical work is recommended to advance the Project to an FS level:

1.22.1 Geotechnical Drilling and Core Logging

- A three-hole geotechnical drilling and rock mass characterization program is proposed to achieve FS data requirements, including targeted drilling of current data voids and areas of interest in the SW-S-SE sectors of the pit, to include discontinuity orientation measurements (where possible), sampling for laboratory strength testing, and televiewer surveys. The holes mainly target waste rock zones outside of the ore zone to determine the geotechnical properties of the units forming the pit walls. Planned metallurgy and infill holes in other areas of the pit could be used as dual-purpose holes for collecting additional "quick / basic" geotechnical data. The core holes should be drilled using a triple tube core barrel to preserve the integrity of the core while drilling and retrieving.
- Laboratory testing is also recommended, including uniaxial compressive strength (UCS) testing (with strain measurements), triaxial strength testing, direct shear testing of discontinuities, and index testing of discontinuity infill materials. Results may be used to confirm or update the geotechnical analysis and slope design parameters provided. Samples should be collected from dedicated (or first-priority) geotechnical drill holes to ensure the appropriate materials are sampled, and to avoid conflicts with exploration sampling and assaying requirements. UCS and triaxial testing should be completed for each of the significant lithological units. The triaxial testing should focus on characterizing the intact rock strength both across and parallel to foliation. Samples of fault or dike contact gouge should also be collected and tested to help characterize the strength of these materials.
- Include the preliminary 3D lithostructural interpretation of all major faults, as viewed in drill core and rock outcrop within 200 m of the pit crest, and integrate them with the regional structural interpretation, into an exploration and geotechnical model for the FS.
- Produce robust 3D digital wireframe models of lithology, alteration with intensity, and structures.
- Update / confirm 2D and 3D stability models and assessments with the aim of optimizing pit slope designs according to anticipated geotechnical performance characteristics.

The following office-based data evaluation tasks are recommended to advance the Project to an FS level:

- Update the existing 3D lithological and/or structural models to incorporate the results of all recent hydro-geotechnical drilling and/or an improved understanding of the deposit geology.
- Interpret structural and geotechnical data and develop a site geologic structural model incorporating major fault and shear structures.
- Consider transient numerical analyses, which take into account the actual mine sequencing.

These studies are estimated to cost a total of CDN$400,000 with CDN$250,000 for drilling and CDN$150,000 for studies, modelling and laboratory work.

1.23 Open Pit Mining

Building on the knowledge and the work completed in the PFS, the following is recommended for advancing the open pit design work to a feasibility-level of study:

- Blasting study
  - further evaluation of pattern sizes and powder factors is required to enhance production and productivity
- Equipment costs and fleet selection
  - update the equipment operating and capital costs from vendors
  - examine alternate vendors for specific equipment
  - optimize the size of haulage trucks
    - current fleet is 17 trucks of 240 t class; further study is needed to determine the most cost effective size
- Examine truck box configurations for tailings haulage to avoid carry back while maximizing carrying capacity of lower density tailings
- Review if autonomous trucks are an opportunity
  - further study recommended
  - technical and social benefits concerns need to be examined
- Ore sampling protocols need to be established
  - definition of ore/waste contacts
  - sample size selection
  - determine if blasthole samples can supplement the proposed Reverse Circulation (RC) samples
- Haul Road design
  - detailed design of north end haul road
  - detailed layout of haul roads for pre-production period
- 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 FS.

1.24 Mineral Processing and Metallurgical Testing

Additional metallurgical testwork should be conducted and should:

- Include diamond drilling done using drill mud additives which assist with core recovery that have been demonstrated to have minimal impact on metallurgical testwork. A bulk sample might be considered to avoid the issue of drilling compound modifying reagents.
- Investigate the impact of drilling mud additives on flotation mass pull with the objective of reducing flotation circuit size and regrind power requirements.
- Further optimize concentrate leach reagents and consider reductions in leach extraction time. This includes reducing the number of concentrate leach adsorption tanks and recover residual gold /silver in solution using the flotation tails CIP circuit.

- Optimize combined tails residual cyanide levels and aim to reduce cyanide detoxification retention time.
- Conduct a full Feasibility Study metallurgical testwork program incorporating variability and production composite testwork. This should include dewatering/filtering tests on the final tailings material.

These studies will require the generation of sufficient tailings to complete the filter testwork. To do this some pilot plant work may be required. The estimated cost is expected to range between CDN$800,000 and CDN$1,200,000.

1.25 Infrastructure

1.25.1 Site Layout

- Optimize the site layout to minimize quarry material requirements.
- Detail the site water handling system with updated hydrogeological and surficial water studies.
- Re-examine the fish habitat to maximize environmental benefit while reducing costs or enhancing closure benefits.

1.25.2 Cofferdam

- Continue site-specific meteorological and hydrology data collection to support refinement of seasonal run-off and design storm estimates. Additional evaluation of long-term regional meteorological data is recommended.
- A model of the Springpole Lake watershed behaviour under baseline (pre-mine) conditions is recommended to support evaluation of lake level fluctuation resulting from dewatering of the open pit area following cofferdam construction, along with lake level fluctuation resulting from design storm events. A baseline watershed model is expected to require monitoring of additional lakes within the Springpole Lake watershed, upstream of the lake outlet.
- Confirm foundation design parameters with additional site investigation, specifically focused on identification of any additional areas of soft lakebed sediments (clay, silt) within the footprint of the cofferdams. Investigation should target additional locations along the alignment of the cofferdams, as well as the upstream and downstream toe of the embankments. Strength parameters must be determined to understand how the lakebed sediments and glacial till will behave under the embankment load (characterize material consolidation and expected strength gain under long-term loading).
- Additional hydrogeological characterization of the bedrock beneath the proposed cofferdams is recommended to support refinement of the extent and depth of bedrock grouting.
- It is recommended that samples of open pit waste and/or samples from potential borrow areas for cofferdam construction be collected for characterization of the rockfill material to be used for construction. Evaluation of the shear strength of the rockfill will confirm design parameters used in stability modelling. Geochemical classification of the proposed material source must be completed to confirm that the rockfill is NAG.

- Additional trade-off of alternative cut-off wall construction methods is recommended, following additional site investigation to evaluate a detailed bedrock profile.
- Additional development of a construction schedule is recommended as the design is advanced.

1.25.3 Waste Storage Facility

- Additional drilling and sampling of the overburden materials is recommended in the area of the WSF to better define the composition and expected variability in material depths. Geotechnical and hydrogeological drill holes are recommended at the maximum embankment sections, along with test pits within both the embankment and basin footprint. Laboratory testing should be completed to refine characterization of the material, including strength and hydraulic parameters for use in stability and seepage analyses.
- Determination of the filtered tailings critical state line is recommended to support selection of an appropriate undrained shear strength ratio for the material (or to support modelling as an undrained material).
- It is understood that a NAG filtered tailings could be produced through removal of a cleaner tailings stream from the filtration circuit. It is recommended that concepts be developed for sub-aqueous storage of the PAG cleaner tailings to improve the overall NP of the WSF.
- Evaluation of the expected chemistry of the surface water run-off from the filtered tailings and PAG waste rock surface and water quality predictions in the water management pond downstream of the WSF will be important for site water management planning (i.e. determination of the requirement for water treatment prior to discharge, if the overall site is in a water surplus).
- Continue site-specific meteorological and hydrology data collection to support refinement of seasonal run-off and design storm estimates. Additional evaluation of long-term meteorological data is recommended.
- A long-term synthetic climate (temperature, precipitation, and evaporation) record for the site will support evaluation of the WSF water balance (along with water quality) based on historic dry and wet periods for the Project area, along with refinement of the overall site water management plan.

The proposed work is expected to cost in the order of:

- ongoing metallurgical monitoring work -CDN$500,000
- hydrometeorological reporting -CDN$50,000
- watershed modelling -CDN$100,000
- site investigation (including drilling) -CDN$1,000,000
- geochemical studies -CDN$250,000
- interface/overlap with EA/permitting -CDN$100,000

The sum of the studies is expected to cost CDN$2M to complete.

1.26 Environmental

Building on the knowledge and the work completed in the PFS, First Mining will move forward with the concurrent federal and provincial EA process. The available environmental data will support this submission, but there are opportunities to add value to the Project generally, and the EA in particular as listed below:

- Store all the project data in an appropriate database.
- Review the historical data in the context of the latest project layouts. Updates as necessary for inclusion in the EA or permitting as appropriate.
- Complete a water quality baseline report, based on data collection to date. This information should be used to provide source terms for water balance modelling.
- Commence continuous snow depth measurements at the Springpole climate station, as well as at least one season of winter snow surveying. Plans for ongoing snow surveying should be incorporated into the project schedule.
- Commence measurement of precipitation as snowfall at the Springpole climate station using an all weather precipitation gauge, or heated rain gauge. Either installation should include a wind screen.
- Continue hydrometric monitoring on creeks around the project area.
- Commence or continue lake level monitoring in nearby lakes potentially impacting or impacted by the Project.
- Completion of a comprehensive hydrometeorological baseline analysis that incorporates both project and regional surface water run-off and climate. This report should serve as a primary source for both mine planning (as described in earlier sections), and water balance modelling, as well as all water related analyses and reporting associated with the EA or permitting as appropriate.
- Complete a comprehensive assessment of the hydrogeological and hydrogeochemical conditions using collected data, with allocation for additional data collection for areas of increased uncertainty.
- Investigate the presence (or absence) of wolverine denning sites.
- Investigate the presence (or absence) of roost trees for Northern Myotes/Little Brown Myotis through a field reconnaissance of suitable habitat areas.
- Develop a stakeholder engagement database.

The above recommended tasks are expected to cost CDN$800,000 to complete. This does not include compilation of the EA but applies to studies that will support the EA or permitting as appropriate.

1.27 Feasibility Study

The level of resource classification, and study work completed to date at the Springpole 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 FS. This work by all the disciplines beyond the previously mentioned studies is estimated to be in the order of CDN$2 M to CDN$3 M.

2 INTRODUCTION

First Mining Gold Corp. (First Mining) is a Canadian exploration and development company, based in Vancouver, British Columbia, Canada, and is publicly-listed on the Toronto Stock Exchange. First Mining is focused on the development of the Springpole Gold Project (the Springpole Gold Project or the Project). First Mining holds a 100% interest in the mineral rights for the Project. The Project is located 110 km northeast of Red Lake, Ontario.

2.1 Terms of Reference

This Technical Report (the Report) and Preliminary Feasibility Study (PFS) was prepared for First Mining by AGP Mining Consultants Inc. (AGP), SRK Consulting (Canada) Inc. (SRK), Knight Piésold Ltd. (Knight Piésold), and Swiftwater Consulting Ltd. (Swiftwater) to present the economic evaluation of the Springpole Gold Project. This Technical Report was prepared for the Springpole Gold Project in accordance with NI 43-101 and Form 43-101F1.

All units of measurement used in this Report and the Mineral Resource and Mineral Reserve estimates are in metric, unless otherwise stated. All grid references are based on the NAD83 Datum (NAD83) UTM coordinate system unless otherwise stated. All currency units are in Canadian dollars (CDN$) 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
Dr. Gilles Arseneau, Ph.D., P.Geo.	SRK - Associate Consultant	1.2 – 1.7,1.9, 4-12, 14, 23, 25.1-25.2, 26.2, 27
Mr. Gordon Zurowski, P.Eng.	AGP - Principal Mining Engineer	1.1, 1.10 – 1.11, 1.13.1, 1.15-1.20, 2,3, 15, 16.1-16.2, 16.5-16.13, 18.1-18.2, 18.5-18.9, 19, 21 (except 21.2.4 and 21.3.3), 22, 24, 25.2, 25.3,25.5.1, 25.6,25.7, 26.1, 26.4, 26.6.1, 26.7, 26.8, 26.9, 27
Mr. Roland Tosney, P.Eng.	AGP - Principal Geotechnical Engineer	16.3,16.4, 26.3, 27
Mr. Cameron McCarthy, P.Eng., P. Geo., P.Tech.	Swiftwater - Principal Engineer	1.14, 20, 26.7, 27
Mr. Duke Reimer, P.Eng.	Knight Piésold - Senior Engineer	1.13.2, 1.13.3,18.3-18.4, 25.5.2, 25.5.3,26.6.2-26.6.3, 27
Dr. Adrian Dance, Ph.D., P.Eng.	SRK – Principal Consultant	1.8,1.12, 13, 17, 21.2.4,21.3.3, 25.4, 26.5, 27

2.3 Site Visits

2.3.1 Geology

Dr. Gilles Arseneau conducted a site visit to the Project from February 10 to February 12, 2012 and again on August 8 and 9, 2012. During these site visits, core logging procedures were reviewed. Several sections of core from the Portage, Camp, and East Extension zones were examined. Sampling procedures and handling were observed. The deposit geology, alteration, and core recovery data were observed for the Portage zone. Mr. Arseneau was fully assisted during the site visits by Springpole personnel and was given full access to data during the site visits.

During the site visits, Dr. Arseneau re-logged mineralized sections of drill core from the Springpole deposit and checked geological units against the recorded written logs. Down-hole survey data entered in the digital database were checked against data entered on paper logs at the site and no errors were noted. Drill site locations could not be verified as most drill sites are situated under a small portion of Springpole Lake, but Dr. Arseneau did observe two drill platforms drilling on the lake during his visit.

2.3.2 Mining

Mr. Gordon Zurowski conducted a site visit to the Project from August 30 to September 1, 2020. The Project site was inspected for two days during the site visit.

While on site, Mr. Zurowski reviewed drill core from the pit area, and visited proposed infrastructure locations including Waste Storage area, cofferdam locations (from land and on water), proposed plant and stockpile locations, and the proposed north haul road.

2.3.3 Site Geotechnical

Mr. Duke Reimer conducted a site visit to the Project from August 30 to September 1, 2020. The Project site was inspected for two days during the site visit.

Mr. Reimer reviewed site-specific information regarding water flows and climatic conditions. Various locations were visited, hiking through brush including the waste storage area, cofferdam locations (from land and on water), proposed plant and stockpile locations, and the proposed north haul road.

2.3.4 Environmental

Mr. Cameron McCarthy conducted a site visit to the Project from August 30 to September 1, 2020. The Project site was inspected for two days during the site visit.

Mr. McCarthy reviewed drill core from the pit and proposed infrastructure locations. He also reviewed site-specific information regarding water flows, climatic conditions, and environmental data collection. Various locations were visited in detail hiking through brush including Waste Storage area, cofferdam locations (from land and on water), proposed plant and stockpile locations, and the proposed north haul road.

2.3.5 Mining Geotechnical

Mr. Roland Tosney conducted a site visit to the Project from August 30 to September 1, 2020. The Project site was inspected for two days during the site visit.

Whilst on site, Mr. Tosney reviewed drill core from the pit area, and reviewed logging procedures. In addition, he independently reviewed the core to establish an understanding of the logging versus his own personal logging to better understand the data and compare it to other projects.

2.4 Effective Date

The Report has multiple effective dates as noted below:

- The effective date of the Mineral Resource estimate for the Project is July 30, 2020.
- The effective date of the Mineral Reserve estimate for the Project is December 30, 2020.
- The effective date of the PFS for the Project is January 20, 2021.

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 First Mining. 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 references section of this Technical Report. This Technical Report is prepared with the most recent information available at the time of study.

2.6 Previous Technical Reports

The Springpole Gold Project has been the subject of several technical reports. The previous NI 43-101 technical reports are listed in the References section of this Technical Report and are summarized in Table 2-2.

Table 2-2: Summary of Previous Technical Reports

Reference	Date	Company	Name
Zabev B., 2004	2004	Gold Canyon	Technical Report on the Springpole Lake Property, Red Lake Mining Division, NW Ontario for Gold Canyon Resources Inc.
Armstrong T., et al…, 2006	2006	P & E Mining Consultants	Technical Report and Resource Estimate on the Springpole Lake Gold Property, Red Lake Mining Division, Northwestern Ontario for Gold Canyon Resources Inc.
Arseneau G, 2012	2012	SRK Consulting	Independent Technical Report for the Springpole Gold Project, NW Ontario, Canada
SRK Consulting, 2013	May 2013	SRK Consulting	Preliminary Economic Assessment for the Springpole Gold Project, Ontario, Canada
SRK Consulting, 2017	Oct. 2017	SRK Consulting	Preliminary Economic Assessment for the Springpole Gold Project, Ontario, Canada
SRK Consulting, 2019	Nov. 2019	SRK Consulting	Preliminary Economic Assessment Update for the Springpole Gold Project for First Mining Gold Corp.

3 RELIANCE ON OTHER EXPERTS

The QPs have relied upon the following other expert reports, which provided information regarding mineral rights, surface rights, property agreements, royalties, and taxation as noted below.

3.1 Ownership, Mineral Tenure and Surface Rights

The QPs have not conducted detailed land status evaluations and have relied upon previous qualified reports and public documents and have reviewed an Opinion provided by Bennett Jones LLP to First Mining dated August 26, 2020 as to the apparent registered owner with respect to certain real property interests and mining claims comprising the Project as of such date, and which is subject to certain assumptions and qualifications as referred therein.

The above referenced document is:

- Title Opinion Letter from Bennett Jones LLP addressed to First Mining Gold. August 26, 2020

This information is used in Section 4 of the Report and in support of the Mineral Resource estimate in Section 14 and the financial analysis in Section 22.

3.2 Taxation

The QPs have not independently reviewed the Project taxation position. The QPs have fully relied upon, and disclaim responsibility for, taxation information derived from experts retained by First Mining for this information.

This information is used in Section 22 of the Report.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Summary

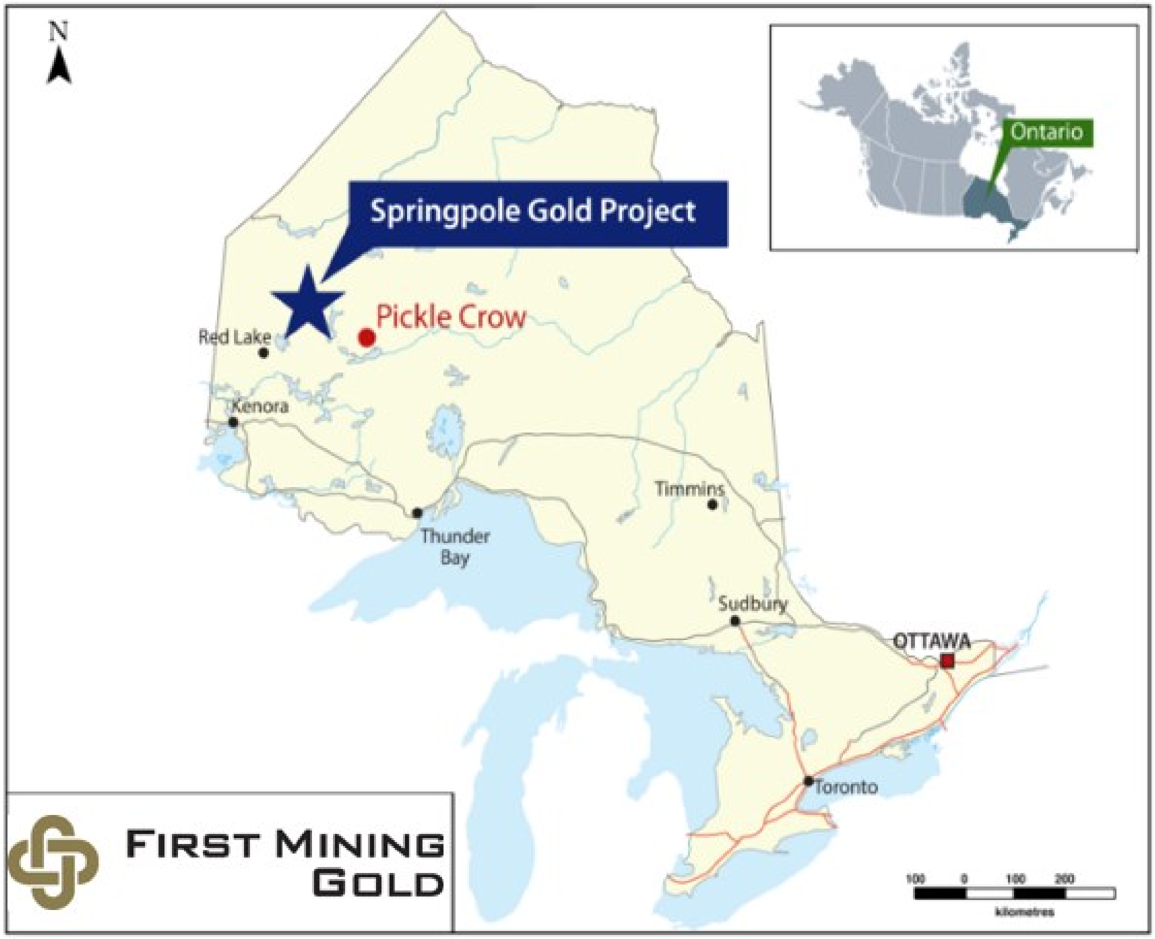
The Springpole Gold Project lies approximately 110 km northeast of the Municipality of Red Lake in northwest Ontario, Canada (Figure 4-1). The Project is centered on a temporary tent-based camp situated on a small land bridge between Springpole Lake and Birch Lake. The latitude and longitude coordinates are:

- Latitude N51° 23′ 44.3″
- Longitude W92° 17′ 37.4″

The Universal Transverse Mercator (UTM) map projection based on the World Geodetic System 1984 (WGS84) zone 15N is:

- Easting 549,183
- Northing 5,693,578
- Average Elevation 395 m

Figure 4-1: Springpole Gold Project Location Map

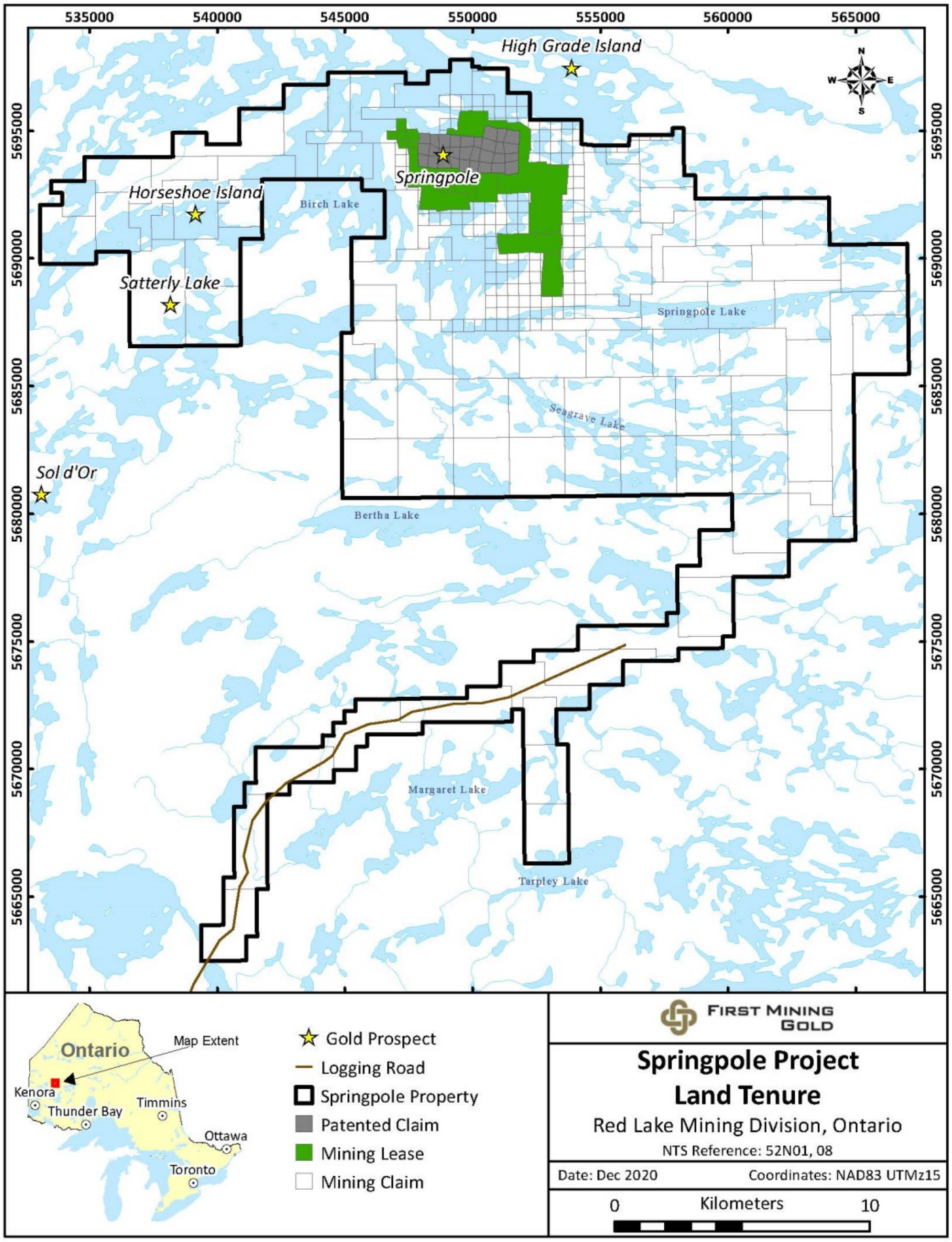


Source: First Mining, 2019

4.2 Land Area

The Springpole Gold Project, wholly-owned and controlled by First Mining, comprises 30 patented mining claims, 282 contiguous mining claims and 13 mining leases totalling an area of 41,943 ha. The overall Springpole Gold Project land area is represented in Figure 4-2 and a more detailed tenure map is shown in Appendix A. A full list of the patents, mining leases and mining claims that comprise the Project is provided in Appendix B.

Figure 4-2: Springpole Gold Project Land Tenure Map



Source: First Mining, 2020

4.3 Mineral Tenure

First Mining acquired 100% of the Springpole Gold Project on November 13, 2015 when it completed the acquisition of Gold Canyon. When the Project was acquired from Gold Canyon, it consisted of 30 patented mining claims and 300 unpatented, contiguous mining claims and six Crown mining leases, totalling an area of approximately 32,448 ha. Additional mining claims were subsequently acquired by First Mining in the Satterly Lake area, and the original unpatented 'legacy' claims were converted into the new Ontario cell claim system in April 2018 (see Section 4.3.4). A further seven mining leases were acquired by Gold Canyon in 2019 by conversion of existing mining claims covering 1,531 ha to mining leases. The area covered by the Springpole Gold Project has increased since 2015 to its current total of 41,943 ha.

With the exception of the 25 patented claims discussed in Sections 4.3.2 and 4.3.3, all mining claims, leases and patents are registered to Gold Canyon, a wholly-owned subsidiary of First Mining.

The QPs have not conducted detailed land status evaluations and have relied upon previous qualified reports and public documents and have reviewed a title opinion provided by Bennett Jones LLP to First Mining dated August 26, 2020 as to apparent registered owner with respect to certain real property interests and mining claims comprising the Project as of the date of such opinion, and which is subject to certain assumptions and qualifications as referred to therein.

4.3.1 Jubilee Gold Claims and Royalty

Gold Canyon had acquired ownership of five patented mining claims in 1993 (11229, 11230, 11231, 12868, and 12869) covering a total area of 96.54 ha from Milestone Exploration Limited, a predecessor entity by way of amalgamation with Jubilee Gold Inc. (Jubilee). These claims are subject to a 3% net smelter returns (NSR) royalty on all minerals mined, produced, and sold from these patented claims, provided the monthly average gold price is USD$700 or more. The NSR was increased to 5%, together with an NSR of 1 - 2.5% on other adjoining properties in which Gold Canyon conducted any mining operations.

In 2010, Gold Canyon renegotiated the applicable NSR on these patented claims with Jubilee. This agreement terminated any applicable royalty on adjoining claims and set the applicable NSR rate payable upon commencement of commercial production at 3% with advance royalty payments of CDN$70,000/yr., adjusted using the yearly Consumer Price Index.

Gold Canyon retained an option to acquire 1% of the NSR for CDN$1,000,000 at any time. As consideration for the renegotiated NSR, it was agreed that previously paid advanced royalties would be forfeited and not credited to any NSR subsequently payable to Jubilee. Gold Canyon paid Jubilee CDN$50,000 and issued 100,000 common shares of Gold Canyon to Jubilee on the date the agreement for the renegotiated NSR was accepted by the TSX Venture Exchange (the Effective Date). In addition, Gold Canyon issued 100,000 common shares to Jubilee on each anniversary date of the Effective Date for the five years that followed. As a result of its acquisition of Gold Canyon, First Mining is subject to this royalty agreement with Jubilee.

First Mining may terminate all royalty obligations by transferring the patented mining claims back to Jubilee. First Mining retains a right of first refusal on any sale of the remaining royalty interest on certain terms and conditions. The five patented claims identified above are fee simple parcels with mining and surface rights attached to all five claims registered with the Land Registry Office, Kenora, Ontario. First Mining has confirmed via independent legal counsel that the five claims have been surveyed, are in good standing, and the property taxes are paid to date.

4.3.2 Leased Claims from R&S Legacy Inc. and Royalty

First Mining, through its wholly-owned subsidiary Gold Canyon, leases 10 patented mining claims (11233-11235, 12896-12901, and 13043) covering a total area of 182.25 Ha. These claims were originally leased to Gold Canyon under the terms of a mineral claims agreement between Gold Canyon and Shirley V. Frahm (Frahm) dated September 22, 2010 (the Original Frahm Agreement). On September 24, 2020, Frahm entered into a purchase and sale agreement with R&S Legacy Inc. (R&S) in furtherance of transferring beneficial ownership to these 10 patented mining claims to her children and grandchildren, and on the same date, Frahm, R&S and Gold Canyon entered into an assignment and assumption of mineral claims agreement (the Assignment Agreement) pursuant to which all of Frahm's rights and obligations under the Original Frahm Agreement were assigned to R&S, and Gold Canyon consented to such assignment. As a result of the Assignment Agreement, the parties to the Original Frahm Agreement are now Gold Canyon and R&S only.

These 10 patented claims are fee simple parcels with all mining and surface rights attached, and registered, together with the notices of lease, with the Land Registry Office in Kenora, Ontario. Under the Original Frahm Agreement, the lease is for a term of 21 years less one day and terminates on April 14, 2031.

On December 11, 2020, First Mining, Gold Canyon and R&S amended certain provisions of the Original Frahm Agreement by entering into a letter agreement (the Amending Agreement). Pursuant to the Amending Agreement, Gold Canyon paid USD$350,000 to R&S as consideration for the amendments, and as a result of the amendments:

- Gold Canyon has the irrevocable option to purchase the 10 patented mining claims from R&S from December 11, 2020 until April 15, 2021 (Purchase Option 1) for USD$7,000,000 (provided that R&S would still retain a 3% NSR on the claims, unless the NSR buy-back right had been exercised by this time), of which USD$1,000,000 may, at Gold Canyon's option, be satisfied by the issuance of common shares of First Mining (First Mining Shares) to R&S.
- Gold Canyon has the irrevocable option to purchase the 10 patented mining claims from R&S from April 16, 2021 until April 15, 2025 (Purchase Option 2) for USD$8,000,000 (provided that R&S would still retain a 3% NSR on the claims, unless the NSR buy-back right had been exercised by this time), of which USD$2,000,000 may, at Gold Canyon's option, be satisfied by the issuance of First Mining Shares to R&S.
- If, on or before April 15, 2025, First Mining provides R&S with written notice, pays USD$250,000 in cash to R&S, and issues 1,000,000 First Mining Shares to R&S, Gold Canyon shall immediately acquire a further irrevocable option to purchase the 10 patented mining claims from April 16, 2025 until April 14, 2031 (Purchase Option 3) for USD$10,000,000, less USD$250,000 (provided that R&S would still retain a 3% NSR on the claims, unless the NSR buy-back right had been exercised by this time). Of the total purchase price, USD$3,000,000 may be satisfied by the issuance of First Mining Shares to R&S.

- If, on or before April 14, 2031, First Mining provides R&S with written notice and pays USD$2,000,000 in cash to R&S, the 21-year term of the lease under the Original Frahm Agreement shall automatically be extended by five additional years and the new expiry date of the lease will be April 14, 2036. In addition, Gold Canyon would immediately acquire a further irrevocable option to purchase the 10 patented mining claims from R&S from April 15, 2031 until April 14, 2036 (Purchase Option 4) for USD$12,000,000 (provided that R&S would still retain a 3% NSR on the claims, unless the NSR buy-back right had been exercised by this time), less USD$2,250,000. Of the total purchase price, USD$4,000,000 may be satisfied by the issuance of First Mining shares to R&S.
- If, on or before April 14, 2036, First Mining provides R&S with written notice and pays a further USD$2,000,000 in cash to R&S, then the term of the lease shall automatically be further extended by five additional years and the new expiry date of the lease will be April 14, 2041. In addition, Gold Canyon would immediately acquire a final irrevocable option to purchase the 10 patented mining claims from April 15, 2036 until April 14, 2041 (Purchase Option 5) for USD$12,000,000 (provided that R&S would still retain a 3% NSR on the claims, unless the NSR buy-back right had been exercised by this time), less USD$4,250,000. Of the total purchase price, USD$4,000,000 may be satisfied by the issuance of First Mining Shares to R&S.
- If at any time during the term of the lease, First Mining commences commercial production, R&S can, by written notice, require Gold Canyon to purchase the 10 patented mining claims for USD$12,000,000 (provided that R&S would still retain a 3% NSR on the claims, unless the NSR buy-back right had been exercised by this time), less any cash payments made by Gold Canyon to R&S in connection with Purchase Option 3, Purchase Option 4, and Purchase Option 5. Of the total purchase price, USD$4,000,000 may be satisfied by the issuance of First Mining shares to R&S.
- If Gold Canyon purchases the 10 patented mining claims from R&S prior to the commencement of commercial production, upon achieving commercial production, Gold Canyon must make a top-up payment to R&S such that R&S would have received an aggregate of USD$12,000,000 from Gold Canyon for the claims (after taking into account any amounts previously paid by Gold Canyon to R&S in connection with the various purchase options). This top-up payment can be any satisfied through any combination of cash payments and First Mining Shares.
- Gold Canyon must pay R&S advance royalty payments on a sliding scale of USD$33,000/year (2010 – 2011), USD$50,000/year (2011 – 2016), USD$60,000/year (2016 – 2021), USD$100,000/year (2021-2031), and USD$120,000/year (2031 – 2041), and all such advance royalty payments shall be deducted from any future NSR payments made by Gold Canyon to R&S.

A 3% NSR with respect to these 10 patented mining claims is payable to R&S upon commencement of commercial production on such claims. During the term of the lease (including if the lease is extended to April 14, 2041), Gold Canyon may, at any time, acquire up to 2% of the NSR for USD$1,000,000 per 1%. In addition, during the term of the lease (including if the lease is extended to April 14, 2041), Gold Canyon has the right to access the 10 patented mining claims to conduct mining operations and produce all ores, minerals, and metals that are or may be found therein or thereon, subject to the small portion of the aggregate surface area that has been reserved for recreational use by R&S.

The remainder of the terms of the Original Frahm Agreement remain unchanged and in full force and effect, including the requirement that Gold Canyon must pay all applicable property taxes related to the 10 patented mining claims during the term of the lease (including if the lease is extended to April 14, 2041), and Gold Canyon maintains a right of first refusal on any sale by R&S of its interest in these 10 patented mining claims on certain terms and conditions set out in the Original Frahm Agreement.

4.3.3 Leased Claims from Springpole Group and Royalty

First Mining has an option and lease to a further 15 patented mining claims (11236, 12867, 12871-12874, 12902-12909) covering a total area of 310.19 ha from a group of individuals and/or companies collectively referred to as the "Springpole Group". These 15 patented claims are fee simple parcels with mining and surface rights attached to all 15 patented claims registered, together with the notice of option and lease, with the Land Registry Office, Kenora, Ontario. The purchase option was originally granted on September 9, 2004 for a term of five years, and the option can be renewed for successive five-year terms by delivering a written renewal notice to the Springpole Group, along with a renewal fee of USD$50,000 and confirmation that at least CDN$300,000 was spent on mining operations in the prior option period. First Mining last renewed the purchase option in September 2019, and the current five-year term of the purchase option expires on September 9, 2023. The current term of the purchase option may be extended by First Mining for further five-year terms by delivering the aforementioned written renewal notice, cash payment and confirmation of expenditures incurred in the prior option period to the Springpole Group.

First Mining is required to make option payments in the aggregate amount of USD$35,000/yr. and to expend an aggregate of CDN$300,000 on mining operations in each option term as a condition of any renewal and to pay all property taxes related to these patented claims. During the option term, First Mining has been granted the exclusive lease, the right and interest to enter upon the 15 patented claims, the right to conduct mining operations, and the right to have quiet possession thereof. First Mining also has the right, at its discretion, to make any use or uses of the 15 patented claims consistent with the foregoing including the construction of roads, railways, conveyors, plants, buildings, and aircraft landing areas, as well as the alteration of the surface of the Project subject to all applicable laws. First Mining has reserved a small portion of the aggregate surface area for the recreational use of a cabin by the members of the Springpole Group.

First Mining holds an option to acquire the 15 claims and would be required to do so upon the commencement of commercial production at any time during the option period by payment of an aggregate of USD$2,000,000. Upon exercise of the purchase option, First Mining must also acquire the cabin on the property for the lesser of fair market value or USD$20,000. A 3% NSR is applicable during the option term upon commencement of commercial production or a 1% NSR if the purchase option is exercised prior to commercial production. First Mining can acquire the remaining 1% NSR by a payment of USD$500,000.

4.3.4 Claims Leased from the Crown (Mining Claims)

In Ontario, Crown Lands are available to licenced prospectors for the purposes of mineral exploration. A licenced prospector must first register a mining claim to gain the exclusive right to prospect on Crown Land. Claims can also be registered in areas where surface rights are not owned by the Crown if the ground is open for staking and mineral rights can be obtained.

Traditional claim staking in Ontario came to an end on January 8, 2018, and on April 10, 2018 the Ontario Ministry converted all existing ground or map-staked mining claims (now referred to as 'legacy' claims) into one or more cell claims or boundary claims as part of their new provincial grid system. The provincial grid is latitude- and longitude-based and is made up of more than 5.2 M cells ranging in size from 17.7 ha in the north to 24 ha in the south.

Under the new provincial system, the 300 legacy claims at the Springpole Gold Project were converted into single cell claims and boundary cell claims on April 10, 2018. Many of these cell claims were subsequently amalgamated into larger multi-cell claims by First Mining, for ease of administration. The current mining claim fabric at the Springpole Gold Project consists of 93 multi-cell claims, 94 boundary cell claims and 95 single cell claims, covering an area of 39,483 ha. A full list of these mining claims including townships, area, claim number, claim type, and claim due date is included in Appendix B, and their locations are shown on the land tenure map in Appendix A. Dispositions (such as mining leases and patents) were not converted and remain as they were. Claim registration is governed by the Ontario Mining Act and is administered through the Ontario Mining Lands Administration System (MLAS) which is the electronic system established by the Minister for this purpose.

4.3.5 Mining Leases

Six mining leases (Leases 108953 to 108958, comprising mining claims KRL562895 to KRL562900) plus one related Crown lease for surface rights were acquired by Gold Canyon from an individual in July 2011 for an aggregate payment of USD$300,000. These claims are subject to a 3% NSR payable upon the commencement of commercial production, with advance royalty payments of USD$50,000/yr. First Mining retained an option to acquire all or a portion of the applicable NSR at a rate of $500,000 per 1% of the NSR at any time. First Mining has permitted the vendor to use a small portion of the property subject to the Crown surface lease, including a vacation home, for recreational purposes provided First Mining was granted a 20-year option to purchase the vacation home for the price determined by an AACI valuator. The vacation home is required to be purchased upon commencement of commercial production.

Subsequent to the acquisition, these mining leases were set to expire. Following the acquisition, Gold Canyon, as the new lessee, renewed all six leases for a further 21 years, effective September 1, 2011.

A further seven mining leases (109846 to 109852) were acquired by Gold Canyon in 2019, by conversion of existing mining claims to mining leases. These leases cover an area of 1,531 ha and have been granted for a 21-year term, effective July 1, 2019.

4.3.6 Claim Maintenance

All mining claims are liable for inspection at any time by the Ministry of Northern Development and Mines of Ontario (MNDM) and may be cancelled for irregularities or fraud in the staking process. Disputes of mining claims by third parties are accepted after one year of the recording date or after the first unit of assessment work is filed and approved. A claim remains valid as long as the claim holder properly completes and files the assessment work as required by the Mining Act and the Minister approves the assessment work.

To keep a mining claim current, the mining claim holder must perform CDN$400 per single cell mining claim unit worth of approved assessment work per year, or CDN$200 per boundary cell mining claim unit, immediately following the initial registration date. The claim holder has two years to file one-year worth of assessment work.

Surface rights are separate from mining rights. Should any method of mining be appropriate, other than those claims for which Crown leases were issued, the surface rights would need to be secured.

4.3.7 Silver Stream with First Majestic Silver Corp.

First Mining, Gold Canyon, and First Majestic Silver Corp. (First Majestic) entered into a silver purchase agreement dated June 10, 2020 (the Silver Purchase Agreement) pursuant to which First Majestic agreed to pay a total of USD$22,500,000 to First Mining over three tranches for the right to purchase 50% of the payable silver produced from Springpole for the life of the Project (the silver stream). First Majestic made the first payment of USD$10,000,000 to First Mining when the transaction closed on July 2, 2020, consisting of USD$2,500,000 in cash and USD$7,500,000 in common shares of First Majestic (First Majestic Shares). A further USD$7,500,000 was paid by First Majestic upon the announcement by First Mining of the completion of the Springpole PFS, with USD$3,750,000 paid in cash and USD$3,750,000 paid in First Majestic Shares. First Majestic will make a final payment of USD$5,000,000 to First Mining upon the earlier receipt by First Mining of approval of a federal or provincial EA for Springpole, with USD$2,500,000 of this amount payable in cash, and the remaining USD$2,500,000 payable in First Majestic Shares.

In addition, in return for the silver stream, following the commencement of production at Springpole, First Majestic will make ongoing cash payments to First Mining equal to 33% of the lesser of the average spot price of silver for the applicable calendar quarter, and the spot price of silver at the time of delivery, subject to a price cap of USD$7.50 per ounce of silver (the price cap). The price cap is subject to annual inflation escalation of 2%, starting at the third anniversary of the commencement of production at Springpole.

Under the terms of the Silver Purchase Agreement, First Mining has the right to repurchase 50% of the Silver Stream for USD$22,500,000 at any time prior to the commencement of production at Springpole (following such repurchase, First Majestic would be left with a right to purchase 25% of payable silver produced from Springpole). First Mining also granted First Majestic a right of first refusal with respect to any future silver stream financings related to Springpole, and First Mining issued 30,000,0000 common share purchase warrants (warrants) to First Majestic on closing of the transaction, with each warrant entitling First Majestic to purchase one First Mining Share at an exercise price of CDN$0.40 for a period of five years.

4.3.8 Royalties Assumptions for Mine Planning and Economic Evaluation

For the purposes of mine planning and economic evaluation, it has been assumed that all buy-back options would be exercised prior to production in accordance with the terms of the agreements summarized in this section.

4.4 Environmental Liabilities

PDAC's Excellence in Environmental Stewardship e-toolkit (PDAC 2009) is used to ensure best practice methods are applied to mineral exploration at the Springpole Gold Project. Improvements to critical areas that affect the environment are underway at all times in an attempt to reduce the environmental footprint of exploration activities. No material environmental liabilities or public hazards associated with the Springpole Gold Project are known to exist on the property. A temporary camp (~0.5 ha) with wood-frame tents was erected for ongoing drilling campaigns. There has been occasional surface clearing related to past drilling work.

4.5 Permits

First Mining complies with permit, notice and consultation requirements as they relate to the ongoing exploration work on the Springpole Gold Project. Legislation that requires material permits and notices include the provincial *Mining Act, Public Lands Act, Lakes and Rivers Improvement Act, Ontario Water Resources Act*, as well as the federal *Fisheries Act*.

Information on permitting required for Project development is provided in Section 20.

4.6 QP Comment on Section 4

To the extent known to the QPs, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the Project that are not discussed in the Report.

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

5.1 Accessibility

During late spring, summer, and early fall, the Springpole Gold Project is accessible by floatplane direct to Springpole Lake or Birch Lake. All fuel, food, and material supplies are flown in from Red Lake or Pickle Lake, Ontario, or from Winnipeg, Manitoba, with flight distances of 110 km, 167 km, and 370 km, respectively. The closest road access at present is 15 km away at the extension of the Wenesaga forestry road.

During winter, an ice road approximately 85 km long is constructed from the South Bay landing point on Confederation Lake to a point about 1 km from the camp. During breakup in spring and freeze-up in fall, access to the Project is by helicopter. Additional winter access may be available via temporary airstrips cleared on nearby frozen lakes.

5.2 Local Resources and Infrastructure

There is no existing infrastructure within 20 km of the Springpole Gold Project area. Businesses in Red Lake, a long-established mining community 110 km to the southwest, provide the majority of the camp's supply needs. The nearest emergency medical facilities are at the Margaret Cochenour Hospital in Red Lake.

The nearest major city is Winnipeg, Manitoba, which is approximately 370 km southwest of the Project and about a 1.3-hour flight by Cessna Caravan.

5.3 Climate and Physiography

January temperatures at the Project range between -40°C and 0°C, and July temperatures range between 20°C and 40°C. Exploration and mining activities can be carried out year-round.

Springpole and Birch Lakes are part of the Albany River system, which flows eastward into Cat River and then northward into Hudson Bay. The Project is underlain by glaciated terrain characteristic of a large part of the Canadian Shield. Land areas are generally of low relief with less than 30 m of local elevation and are separated by a series of interconnected, shallow lakes.

Tree cover consists of mature spruce, balsam, birch, and poplar. Black spruce and muskeg swamps occupy low-lying areas. Glacial till is generally less than 1 m in thickness. Outcrops are limited and small and are generally covered by a thick layer of moss or muskeg.

6 HISTORY

The history of the Springpole Gold Project prior to 2006 is excerpted from the Technical Report and Resource Estimate on the Springpole Lake Gold Property (Armstrong et al…, 2006).

Gold exploration was carried out during two main periods, one during the 1920s to 1940s, and a second period from 1985 to the present.

In 1925, the discovery of gold at Red Lake brought prospectors into the Springpole Lake area. Visible gold in outcrop on the Project was first discovered north of the Birch Springpole Lake portage and prospected by Northern Aerial Mineral Exploration Ltd. in 1928 (Harding, 1936). The showing was initially covered with eight claims around 1933 by prospector Tom Dunkin, who then completed the first stripping and shallow trenching in 1934.

Between 1933 and 1936, the Windigokan Sturgeon Mining Syndicate conducted extensive trenching and prospecting, including 10 short holes totalling 458.5 m. The claims were then transferred to Springpole Mines Ltd. who carried out limited trenching and prospecting in 1945.

The Casey Summit Mine (later renamed the Casummit Mine), approximately 10 km to the north, started operation around this time. This mine ultimately produced 101,975 oz of gold and 9,788 oz of silver (Beakhouse, 1990) and is the only significant past producer of precious metals in the Birch-Springpole Lake area.

This early prospecting activity and production from the Casummit Mine region prompted a more detailed geological investigation of the vicinity by the Ontario Department of Mines. The Birch Lake area was mapped at a scale of 1:63,360 by Harding (1936).

Reconnaissance-style mapping of the Birch Springpole Lake area has since been repeated four times:

1. To study volcanic characteristics of selected Superior Province greenstone belts (Goodwin, 1967)
2. To extend volcanic stratigraphy hosting the South Bay base metal mine into the Springpole area (Thurston et al..., 1981)
3. To stimulate gold exploration in the area after closure of several mines near Red Lake (Good et al…, 1988)
4. To study the stratigraphy of epiclastic and volcaniclastic facies units, northern Birch-Uchi greenstone belt (Devaney, 2001a)

The area remained dormant until 1985 when Gold Fields Canadian Mining Ltd. (GFCM) optioned the Frahm claims and, in 1986, the Milestone claims and Maple Leaf (now Springpole Group) claims. GFCM conducted an airborne (Aerodat) geophysical survey in 1985 over the entire claim group. On the 30 patented mining claims (Frahm, Milestone, and Springpole Group), line cutting was done at both 30.5 m (100 ft) centres (Milestone claims) and 61 m (200 ft) centres (Frahm and Springpole Group claims). Subsequently, geological mapping, humus geochemistry, and ground geophysics (very low frequency (VLF), magnetic (Mag), and IP) were conducted over the grids.

From 1986 through 1989, GFCM completed 118 diamond drill holes in seven drill phases totalling 38,349 m. In addition, during 1986 and 1987, approximately 116,119 m^2 of mechanical stripping was carried out by GFCM, and four petrographic reports were produced. As a result of this work, GFCM identified several gold -bearing zones on the Project site that included:

- Portage zone, entirely under a portion of Springpole Lake, but the largest of the zones and therefore the main focus of the bulk of the exploration work
- Jasper zone, a deep narrow higher-grade zone in a banded iron formation horizon
- several smaller but higher-grade zones on the land portion of the Project and close to surface, including the Main zone, Vein zone, Hillside zone, Camp zone, North Porphyry zone and East Extension zone

Late in 1989, GFCM entered into a 50/50 joint venture with the combined interests of Noranda and Akiko-Lori.

From 1989 through 1992, Noranda conducted an IP survey over the central portion of the Portage zone and tested the property with eighteen core holes totalling 5,993 m. The majority of the drilling was conducted on the Portage zone.

At the same time, and under a separate option agreement with BP Resources Canada, Noranda completed a seven-core hole drill program around the east margins of Springpole Lake on claims then owned by BP Resources. BP Resources in turn completed lake-bottom sediment sampling of Springpole Lake east of Johnson Island.

In 1992, Noranda dropped its interest in the property leaving Akiko-Lori to carry out further exploration while carrying its 50% partnership with GFCM. During 1992 to 1994, Akiko-Lori/Akiko Gold completed an additional 15 diamond drill holes totalling 5,154 m.

By 1995, Akiko Gold was reorganized into Gold Canyon and GFCM's interest was acquired by Santa Fe Mining as part of an asset exchange with London based Hanson Plc., which controlled GFCM. During 1995, a joint venture between Gold Canyon and Santa Fe carried out an exploration program consisting of remapping of the main area, of some of the existing drill core, and a reinterpretation of the geology.

During the 1995 and 1996 programs, Santa Fe drilled an additional 69 holes totalling 15,085 m on the Springpole Gold Project proper and two drill holes on Johnson Island. By late 1996, the takeover of Santa Fe by Newmont Gold Company was nearing completion. Just prior to the merger with Newmont, Santa Fe exchanged their 50% interest in the property for a tax credit that left Gold Canyon with a 100% ownership. After Santa Fe’s departure, Gold Canyon continued exploration at the Springpole Gold Project in 1997 and 1998 with another 52 core holes totalling 5,643 m.

Paso Rico had an option to earn an interest in the Project and, in the summer of 1998, conducted with Gold Canyon a lake bottom sediment sampling program in several areas of Springpole Lake. The results of this survey identified several follow-up targets that were tested in 1999 by Paso Rico with 12 core holes totalling 2,779 m. In 2000, Paso Rico withdrew from the Project leaving Gold Canyon with its current 100% interest.

During 2004, 2005 and 2006, diamond drilling programs were conducted on the property by Gold Canyon. A total of 109 holes were completed during this period, over 17,322 m.

A summary of the historical drilling completed on the Property between 1986 and 2006 is provided in Table 6-1.

Table 6-1: Summary of Historical Drilling at Springpole 1986-2006

Diamond Drill Hole	Company	Period	Number of Holes	Metres Drilled
BL-1 to BL-124 SP-1 & SP-2	Goldfields Canadian Mining Ltd.	1986-1989	118	38,349
BL-125 to BL-141, OB-1 incl. 4 ext.	Noranda / Akiko JV	1990-1991	22	5,993
SP-01 to SP-09	Akiko-Lori Gold Resources Ltd.	1992	9	2,088
BL-142 to BL-147	Akiko Gold Resources Ltd.	1993-1994	6	3,066
BL-148 to BL-216	Santa Fe Canadian / Gold Canyon Resources Inc. JV	1995-1996	69	15,085
BL-271 to BL-248 incl. 1 ext. hole	Gold Canyon Resources Inc.	1997	33	3,593
BL-249 to BL-267	Gold Canyon Resources Inc.	1998	19	2,050
BL-268 to BL-279	Paso Rico	1999	12	2,779
BL-280 to BL-320 Incl. 5 holes ext.	Gold Canyon Resources Inc.	2004	46	4,799
BL-321 to BL-352 BL-209 twinned, BL-30 and BL-209 wedges	Gold Canyon Resources Inc.	2005	42	9,364
BL-353 to BL-373	Gold Canyon Resources Inc.	2006	21	3,159

6.1 Fall 2007 Program

In the fall of 2007, Gold Canyon embarked on a limited exploration program to further investigate the Fluorite zone that was identified by Noranda during its trenching program in 1990. Noranda identified the potential for Ontario's largest undeveloped fluorite deposit in the form of a sovite (calcitic carbonatite) from four trenches and having over 850 m of strike with high grade values up to 35.6% fluorite (CaF_2).

During the course of the program, 46 one-metre samples were collected from four "cuts" across a previously identified 23 metre-wide zone of fluorite mineralization at the western end of Long Skinny Pond — a thin, narrow pond to the north of camp that channels water from Birch Lake to Round Pond and into Springpole Lake via a narrow stream channel.

Sampling results were inconclusive as fluorite content (CaF_2) was not analyzed. Additionally, the samples were tested for their rare earth element potential, but these results were also inconclusive. Gold values were borderline anomalous and did not warrant any follow up.

6.2 Summer–Fall 2009 Program

From early August through to the end of October 2009, Gold Canyon embarked on a core re-logging and re-sampling program. Five geologists, under the supervision of Jeff Chambers, a senior consulting geologist, re-logged and re-sampled a portion of the historical drill core stored at the Project site and temporary tent camp.

A total of 417 diamond drill holes were completed on the Springpole Gold Project prior to 2009; drilling had begun in 1933 (Zabev, 2004). This amounted to a total of approximately 98,262 m of core drilled. Unfortunately, not all the drill core is on-site. The 1933 through 1936 drill holes 1 to 10 are missing. Also missing are drill holes BL-20 through BL-53 completed by the GFCM exploration program from 1986 to 1988. From drill log records, it appears the whole cores were sent for analysis. Drill hole BL-95A, which is an extension of drill hole BL-95 that was completed during the Noranda program in early 1990, is also missing. In addition to missing holes, there are many intervals throughout the core inventory that are missing.

At the time the re-logging and re-sampling program was conducted, the full database of available historical core logs and historical assay data was not fully compiled and was not available to the geologists working in the field. The dataset used in the field was a compilation from the database that was compiled as a result of the work carried out for the 2006 technical report (Armstrong et al..., 2006).

6.3 Core Re-Logging Program

A total of 115 drill holes were re-logged during the fall 2009 program, which equates to approximately 31% of the 374 drill holes that were believed to be on the property at that time. Forty-nine drill holes are known to be missing, and the above count does not include the numerous mineralized intervals that are missing within drill holes and were not re-logged.

Core re-logging was carried out in a summary format designed to be easily incorporated into later modelling efforts. This meant drill holes were divided into broad units based upon average lithology, alteration, and mineralization. Quality of logging varied between geologists, as it was clear that a formal standard for logging was not adopted. Logging efforts were further hampered by core intervals that contained little, if any, useful material due to sampling of all or nearly all of the recovered core, as well as degradation and decay of core boxes and core racks.

The information obtained from the re-logging exercise was used to plan the phased drill program of 2010 to 2012. All re-logged core forms were scanned and now form a part of the digital database stored at First Mining's office in Vancouver, British Columbia.

At the end of the core re-logging program, several days were taken to examine drill core from critical areas. The top 6 to 12 m (20 to 40 ft) of core was examined briefly, and a simplified lithology was assigned. Overburden was excluded. The intent of the exercise was to apply the noted lithology to produce a crude geologic map. This could then be used to assess the outline geometry of the trachyte intrusive, and all the associated breccia phases.

For the re-sampling program, a total of 2,580 samples were taken from the historical drill core. This included 132 standards, blanks, and duplicates, totalling approximately 5% of the number of samples collected. All samples were taken from drill core that was re-sampled by cutting the remaining drill core in half. This resulted in either a half or a quarter of the core remaining, depending on whether the interval had been sampled originally. Due to the small core diameter, core was not cut to less than one-quarter to preserve material for future reference. Table 6-2 represents significant intercepts from historical drilling combined with the re-sampling work outlined here.

At the end of the core re-sampling program, 14 samples for thin-section petrographic analysis and 3 samples for mineral petrographic analysis were collected. The samples collected were deemed representative of the principal lithologies occurring across the Springpole Gold Project.

6.4 Acquisition by First Mining Gold Corp.

On November 13, 2015, First Mining (which was called First Mining Finance Corp. at the time) completed the acquisition of Gold Canyon, and as a result, acquired the Springpole Gold Project.

Table 6-2: Historically Significant Intercepts from 2009 Re-Sampling Program

Main Zone					Portage Zone					East Extension & Main Zone				
Drill Hole	From (m)	To (m)	Interval (m)	Au Grade (g/t)	Drill Hole	From (m)	To (m)	Interval (m)	Au Grade (g/t)	Drill Hole	From (m)	To (m)	Interval (m)	Au Grade (g/t)
01	22.48	27.14	4.65	6.04	92-01	100.43	118.44	18.01	3.72	BL12	25.92	38.72	12.80	1.85
BL1	43.90	53.96	10.06	4.57	92-04	194.66	204.88	10.22	7.11	BL115	99.38	110.98	11.59	2.73
BL102	42.38	49.08	6.70	11.60	92-06	175.45	191.80	16.34	5.58	BL162	35.98	56.41	20.43	1.15
BL103	29.27	34.45	5.18	2.44	BL100	158.23	178.66	20.43	3.53	BL163	7.10	28.05	20.95	4.78
BL11	214.63	224.09	9.45	6.53	BL121	104.91	140.55	35.64	7.57	incl	16.16	28.05	11.89	7.92
BL11	295.42	317.38	21.96	1.75	BL122	163.41	241.16	77.75	1.57	BL163	88.20	102.44	14.23	2.07
BL157	60.68	62.20	1.52	206.74	incl	166.77	177.13	10.36	6.70	BL165	9.45	39.94	30.49	2.92
BL160	16.46	26.53	10.06	16.19	BL125	110.13	117.53	7.40	2.41	incl	17.98	33.23	15.24	4.38
BL161	4.48	25.61	21.13	3.61	BL125	150.74	158.74	8.00	5.68	BL166	10.36	24.39	14.02	1.42
BL183	105.80	119.82	14.02	1.33	BL126	104.33	120.13	15.80	2.60	BL168	72.26	87.50	15.24	1.60
BL190	110.06	111.28	1.22	15.70	BL127	123.53	131.73	8.20	7.07	BL172	18.14	39.63	21.49	10.44
BL197	40.30	54.60	14.30	1.54	BL128	174.04	211.65	37.61	2.13	incl	25.92	29.27	3.35	50.50
BL198	87.68	99.99	12.31	1.52	BL129	139.04	185.05	46.01	1.57	BL202	40.24	68.97	28.73	1.73
BL209	455.48	456.34	0.85	182.06	BL131	91.32	238.26	146.94	1.09	BL204	44.82	53.96	9.14	20.53
BL23	77.65	87.50	9.85	9.60	Incl	199.05	214.05	15.00	2.06	incl	45.73	47.16	1.43	136.58
incl	86.89	87.50	0.61	109.37	BL132	234.66	258.97	24.31	2.06	BL217	14.66	42.07	27.41	14.96
BL25	200.97	233.54	32.57	1.66	BL26	93.29	154.27	60.98	2.29	incl	14.66	15.24	0.58	46.18
BL264	5.18	45.73	40.55	4.56	BL308	154.76	179.27	24.51	1.29	incl	19.55	22.26	2.71	39.08
incl	5.18	13.72	8.54	7.04	BL308	214.45	321.34	106.89	2.35	incl	35.06	42.07	7.01	35.37
incl	34.39	42.56	8.17	8.96	Incl	225.06	241.49	16.43	5.81	BL220	16.25	54.52	38.26	3.54
BL280	15.85	23.14	7.28	4.59	BL310	98.08	118.54	20.46	1.77	incl	16.25	17.38	1.12	54.17
BL282D	95.70	97.41	1.70	17.84	BL310	136.62	151.53	14.91	2.91	BL221	2.13	17.34	15.21	2.92
BL285D	20.63	26.53	5.89	2.23	BL311	133.23	145.43	12.19	2.48	BL222	3.66	28.66	25.00	5.85
BL3	4.27	55.48	51.21	2.14	Incl	134.75	137.49	2.74	6.91	incl	17.38	18.76	1.38	73.03
incl	45.12	50.00	4.88	14.87	BL312	36.89	66.16	29.27	1.43	BL225	3.05	26.22	23.16	2.66
BL300	45.73	49.69	3.96	4.01	BL33	258.94	274.69	15.75	1.22	incl	21.95	26.22	4.26	12.13
BL302	27.44	31.53	4.09	3.73	BL41	110.37	134.63	24.27	2.70	227	87.95	92.98	5.03	5.25
BL303	44.82	63.94	19.12	5.00	BL41	164.63	282.92	118.29	1.64	BL228	43.29	67.84	24.55	18.63
BL305	67.56	68.90	1.34	23.87	Incl	233.84	263.11	29.27	2.92	incl	65.25	66.16	0.91	120.99
BL306	23.88	63.85	39.97	1.01	Incl	235.67	242.99	7.32	5.15	BL292	20.63	40.67	20.04	10.28
incl	33.84	38.29	4.45	4.76	BL42	101.52	127.44	25.92	1.05	incl	37.62	39.24	1.62	49.99
BL307	16.31	49.78	33.47	1.21	BL67	196.95	218.30	21.34	2.11	BL296	53.72	103.54	49.81	3.87
BL354	85.03	85.79	0.76	30.31	BL69	216.77	219.82	3.05	21.07	incl	59.40	59.84	0.45	102.00
BL356	36.89	40.91	4.02	31.67	BL79	248.78	253.35	4.57	6.09	incl	63.91	65.37	1.46	47.85
incl	39.94	40.91	0.97	127.13	BL80	448.47	460.67	12.20	2.52	incl	85.97	87.50	1.53	32.16
BL68	150.15	284.36	134.21	1.41	BL85	344.59	380.80	36.21	1.40	BL328	8.99	54.02	45.03	3.25
incl	150.15	181.16	31.01	1.88	BL88	297.52	350.91	53.38	1.97	incl	41.34	49.69	8.35	8.61
incl	217.98	243.90	25.92	2.30	BL90	65.86	76.04	10.18	3.92	BL330	33.84	34.45	0.61	16.59
BL7	50.61	68.90	18.28	1.57	BL93	169.20	178.66	9.45	2.25	BL336	173.72	174.60	0.88	14.02
BL9	25.00	37.20	12.20	4.23	BL94	264.03	271.65	7.62	2.09	BL340	25.46	39.97	14.51	15.54
incl	28.87	34.76	5.89	7.18	BL95	396.04	417.11	21.07	1.66	incl	35.34	39.97	4.63	43.64
BL96	39.63	58.71	19.08	2.89	BL99	198.47	314.33	115.86	1.53	BL343	25.70	56.13	30.43	4.33
incl	53.56	58.72	5.15	8.49						incl	25.70	29.64	3.94	27.38
BL98	39.94	73.48	33.54	1.16										

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The following excerpt is quoted from Devaney (2001b) and provides the most concise geologic description of the regional geology of the Springpole-Birch Lake area:

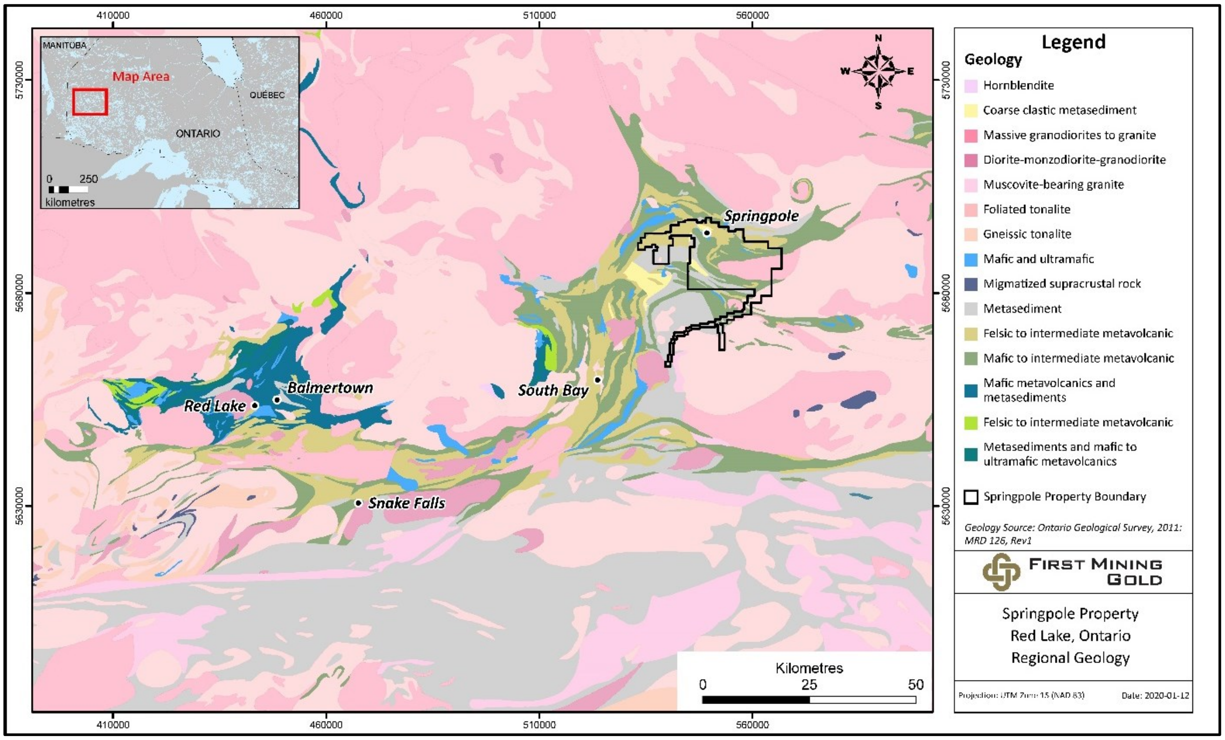
> *"The Birch-Uchi Greenstone Belt is the portion of the Uchi Sub-province with an arcuate, concave to the southeast, (i.e., a major oroclinal bend between the Red Lake and Meen-Dempster portions of the sub-province). Studies of the southern part of the Birch-Uchi greenstone belt as a rootless greenstone belt only a few kilometres thick, have revealed a long (ca. 3.0 to 2.7 Ga), multistage history of crustal development. Based on mapping, lithogeochemistry, and radiometric dating, the supracrustal rocks of the greenstone belt were subdivided into three stratigraphic group-scale units (listed in decreasing age): the Balmer, Woman and Confederation assemblages. This three-part subdivision was applied to most of the Uchi Subprovince. The Confederation assemblage is thought to be a continental margin (Andean-type) arc succession, versus the less certain tectono-stratigraphic context of the other assemblages. Workers performing recent and ongoing studies of the southern Birch-Uchi greenstone belt and the Red Lake greenstone belt (i.e., the Western Uchi Subprovince NATMAP Project) have proposed some modifications and additions to the Balmer-Woman-Confederation stratigraphic scheme. As discussed herein, some relatively small conglomeratic units likely form a synorogenic, discontinuously distributed, post-Confederation assemblage in the Birch-Uchi greenstone belt. Radiometrically dated plutons within the Birch-Uchi greenstone belt are of post-Confederation assemblage, ca. 2725-2700 Ma age.*
>
> *The northern margin of the Birch-Uchi greenstone belt forms a pattern of sub-regional scale cusps of supracrustal strata alternating with batholiths. Basaltic units are prominent around the periphery of the greenstone belt and may be part of the Woman assemblage, but the accuracy of this stratigraphic assignment is unknown. Based on a ca. 2740 Ma age of Shabumeni Lake [intermediate to felsic fragmental] volcanic rocks at a site near the northern greenstone belt margin, suggested that Confederation assemblage age rocks make up the bulk of the greenstone belt".*

The regional geological setting of the Springpole Gold Project is illustrated in Figure 7-1.

It is noteworthy that in many of the regional geology descriptions of the Birch-Uchi Greenstone Belt, especially those in the vicinity of Springpole and Birch Lakes, the structural geology is poorly understood. Many authors make relatively brief mention of the complexities that dominate the geology and geomorphology of the low-lying areas. However, the Archean Orogenic gold deposit model developed by various authors has been applied to the mineral deposits of the Archean Superior Province. Recent concise summaries of these orogenic gold deposits can be found in: Groves et al…, (1998), Hagemann and Cassidy (2000), Aufarb et al…, (2005), and Robert et al…, (2005).

Orogenic gold deposits are epigenetic, structurally controlled gold deposits that are hosted in orogenic belts. They are generally accepted as having formed during late stages of continental collision. Most of the discovered orogenic gold deposits in the world occur in greenstone belts situated on the margins or within Archean cratons in North America, Australia, and southern Africa.

Figure 7-1: Springpole Gold Project – Regional Geology



Source: First Mining, 2020

7.2 Property Geology

The Springpole Gold Project has been extensively studied during past programs and the findings of those studies will not be covered in detail here; however, they are adequately covered in the technical reports of Zabev (2004) and Armstrong et al..., (2006).

The following subsections summarize the geology interpreted from field observations and petrographic analysis of drill core from the 2009 re-logging program and from drill core produced during the 2010 and 2011 programs.

7.2.1 Trachyte Porphyry Intrusion

A polyphase alkali, trachyte intrusion displaying autolithic breccia textures lies at the heart of the Springpole Gold Project. The intrusion is comprised of a system of multiple phases of trachyte believed to be part of the roof zone of a larger syenite intrusion, as fragments displaying phaneritic textures were observed from deeper drill cores in the southeast portion of the Portage zone. Early intrusive phases consist of megacrystic feldspar phenocrysts, up to 5 cm long, of albite and orthoclase feldspar in an aphanitic groundmass. Successive phases show progressively finer grained porphyritic texture while the final intrusive phases are aphanitic.

In 2009 and 2010, Gold Canyon carried out petrographic studies (Saunders and McIntosh 2009, 2010) of historical drill core and drill core from the drill holes SP10-001 through SP11-006. The study confirmed the trachyte intrusion is the dominant lithology within the Project area and is a host to mineralization. Interpretation of the intrusive complex is complicated by a mixture of overprinted regional and local metamorphic events related to burial and tectonism.

Pervasive alteration and metamorphism have reduced the original porphyry intrusion to a complex alteration assemblage dominated by sericite, biotite, pyrite, calcite/dolomite, and quartz. Primary igneous textures are remarkably well preserved in places and give indications to the possible genesis of the initial phase of gold mineralization. Within the country rocks to the north and east are trachyte and lamprophyre dikes and sills that source from the trachyte - or syenite-porphyry intrusive system.

7.2.2 Confederation Age Volcanic and Siliciclastic Rocks

The country rocks pre-date the alkali intrusion and are composed of a complex sequence of altered and metamorphosed intermediate andesitic volcanic and associated volcaniclastic rocks, siliciclastic sedimentary rocks, chemical sediments including banded iron formation (BIF), and coarse pebble conglomerates. Devaney (2001a) indicates that the sediments are likely of the Confederation assemblage dating at around 2,740 Ma, representing the proximal portions of a mixed volcanic-sedimentary basin.

7.2.3 "Timiskaming-type" Conglomerates

Barron (1996) states pebble conglomerate outcrops between Springpole Lake and Birch Lake contain clasts of the trachyte porphyry, suggesting that the "Timiskaming-type" conglomerates postdate the intrusion. Devaney (2001a) suggests these arcuate form conglomerates represent late orogenic, deformed, dextral sense strike-slip (pull-apart) basins of "Timiskaming-type", late Archean, post Confederation assemblage age rocks.

7.3 Structure

Deformation has added complexity to the apparent geometry of, and the potential of, the Springpole gold deposit. Gravity and magnetic surveys carried out across the Project demonstrate that several phases of deformation are evident. Banded iron formations describe north-northwest facing tight to isoclinal antiforms and synforms and are illustrated on the geological map produced during the summer 2005 mapping program (Armstrong et al..., 2006) and are evident as strong magnetic anomalies on the aeromagnetic surveys conducted by Fugro.

In 2011, SRK was contracted to carry out a preliminary study of the structural controls on mineralized deposit geometry. The study found the deposit was subjected to several deformational events including, but not limited to:

- early folding resulting in tight to isoclinal fold geometries and development of associated shear zones
- intermediate large scale, potentially deep-rooted shear zones
- late stage brittle faulting

Further study is required to definitively establish the relationship of the timing of deformational events with respect to economic mineralization.

7.4 Alteration

All rocks on the Project exhibit pervasive alteration that consists of multiple overprinted phases. Distinguishing between the individual phases will take considerable study on a microscopic scale. The country rocks and alkali intrusive rocks exhibit pervasive green-schist facies metamorphism and alteration, probably the result of burial. This manifests as chlorite, calcite, and pyrite in the intermediate volcanic rocks, pyritization of the banded iron formation, and sericite-pyrite alteration within the alkali intrusive associated rocks.

Studies conducted as a part of the exploration work carried out from the fall of 2009 and the winter/spring of 2010 show there is evidence of early alteration phases. These probably resulted from magmatic hydrothermal fluids associated with porphyry gold mineralization and the associated epithermal/mesothermal style gold mineralization. This occurs as potassic and phyllic/sericitic alteration: K-feldspar, biotite, and muscovite (sericite), respectively, and is nearly pervasive in the alkali intrusive rocks and surrounding country rocks. Regional metamorphism has subsequently altered the primary hydrothermal mineral assemblages, but textures have been preserved with the exception of areas of high strain (e.g., northwest trending shear zones).

Advanced argillic alteration appears throughout the trachyte intrusive and occurs in some of the late stage lamprophyre dikes though on a small scale. It is difficult to assess at what stage argillic alteration occurs, but it appears to define an envelope around the Portage zone potassic-alteration/mineralization, suggesting an origin more in keeping with zoned alteration associated with epithermal-style porphyry intrusive hosted gold deposits.

7.5 Mineralization

7.5.1 Porphyry-style Mineralization

The main intrusive complex appears to contain many of the characteristics of alkaline, porphyry-style mineralization associated with diatreme breccias (e.g., Cripple Creek, Colorado). Direct comparison with drill core from the two sites shows a number of consistent textures and styles of mineralization. A recent observation made from drilling, combined with the airborne magnetic survey, shows that potentially economic gold mineralization is coincident with an unexplained geophysical anomaly. This style of mineralization is characterized by the Portage zone and portions of the East Extension zone where mineralization is hosted by diatreme breccia in aphanitic trachyte. It is suspected that ductile shearing and brittle faulting have played a significant role in redistributing structurally controlled blocks of the mineralized rock. Still to be identified is a form of porphyry style alteration zoning consisting of an outer zone of phyllic (sericite) dominant alteration with narrow zones of advanced argillic alteration characterized by illite and kaolinite, and a core zone of intense potassic alteration characterized by biotite and K-feldspar.

Multi-element analysis conducted during the 1992 program on the Portage zone, combined with gold assays, gave the first indication of the style of mineralization at Springpole. Diamond drilling in the winter of 2010 revealed a more complex alteration with broader, intense zones of potassic alteration replacing the original rock mass with biotite and pyrite. The expected alteration zone envelopes or shells are very difficult to define due to complex sheared geometry and poorly defined contact zones of the deposit. In the core area of the deposit where fine-grained, disseminated gold mineralization occurs with biotite, the primary potassic alteration mineral, gold, displays a good correlation with potassium/rubidium.

7.5.2 Lode Gold Mineralization

The intrusion of the trachyte complex into the volcanic pile, as well as the chemical and siliciclastic sedimentary rocks in a near surface environment, produced mesothermal to epithermal style lode vein mineralization. The difference between mesothermal and epithermal mineralization regimes is the temperature and pressure of the mineralizing fluids.

Higher temperature (mesothermal) fluids would have existed within the emplaced intrusive, associated with the diatreme breccias, and in the immediately adjacent wall rock/country rocks. In the porphyry intrusive, and at the contact between intrusive and wall rock in the East Extension zone, and localized within the Main zone, mesothermal style quartz-biotite-calcite-sulphide veins with occasional tourmaline are observed with occasional coarse, visible gold.

Further from the intrusive complex and wall rock contact zones, where meteoric fluids have a greater influence, epithermal style vein textures and mineralization styles dominate. These consist of banded to sucrosic quartz calcite veins with a lower temperature mineral assemblage including sericite, minor biotite, possible adularia, calcite, dolomite, and ankerite; here gold, silver, and tellurium alloys dominate, including electrum and gold -silver tellurides.

7.5.3 Gold Remobilization During Metamorphism

As evidenced from the high degree of deformation, both ductile and brittle—in the form of isoclinal folding, ductile shear zones with protomylonite and blastomylonite textures, and brittle fault textures — the Springpole deposit has been subjected to alteration and metamorphism. These processes alone have remobilized gold in epithermal quartz veins that were the principal motivation for exploration at the Springpole Gold Project in the late 1980s and early 1990s, when shear zone hosted gold deposits were the targets of choice in the Red Lake area.

8 DEPOSIT TYPES

Mineralization at the Springpole Gold Project is dominated by large tonnage, low grade, disseminated porphyry-style or epithermal-style gold mineralization associated with the emplacement of an alkali trachyte intrusion. Textures observed in the extensive repository of drill core appear to confirm that the disseminated gold-silver-sulphide mineralization, the mesothermal to epithermal lode vein gold mineralization, and the banded iron-formation hosted gold mineralization are all the result of the emplacement of multiple phases of trachyte porphyry and associated diatreme breccias, hydrothermal breccias, dikes and sills.

The initial exploration on the property was conducted on the assumption the mineralization was a typical example of Archean mesothermal, sulphide-hosted lode gold type. While this model has not been completely ruled out, it has been replaced in favor of a high-level emplacement porphyry model. Barron's thesis (1996) work presented strong evidence that the gold and associated fluorite mineralization at Springpole are genetically related to the high-level emplacement of a large, alkaline porphyry intrusive and breccia pipe complex.

Barron considered the Springpole Complex to be the end product of magmatic fractionation processes and of fluids that evolved from magmatic to hydrothermal in the high level, sub-volcanic porphyry environment. These processes produced a low-grade gold-porphyry-epithermal type deposit and associated high-grade veins and breccia pipes.

Santa Fe geologists felt the nature of the mineralization at Springpole had many similarities with deposits of the Cripple Creek District, Colorado, including the Cresson Mine. Detailed mapping on the land-based portions of the property by Santa Fe geologists showed that most, if not all, of the gold mineralization at the Springpole Gold Project is spatially associated with the feldspar porphyry diatreme dikes, veins, and diatreme breccia. The following is a brief description of this model in the Springpole area.

8.1 Depositional Environment

Based upon the abundance and size of epizonal trachyte porphyry intrusive masses and the widespread brecciation and alteration centered on the Portage zone, Barron (1996) considered this area to be the apex of a buried syenite stock. A high emplacement level for the Portage zone and surrounding porphyry is further supported by the lack of contact metamorphic effects in the enclosing country rocks. Trachyte clasts within the basal conglomerate overlying the intrusive complex indicate it was subjected to surface erosion.

The rarity of trachyte clasts and their restriction to the base of the conglomerate unit would seem to indicate erosion over a short time interval. The lack of voluminous trachyte flows suggests there was no markedly positive volcanic edifice. Barron (1996) concluded that collectively these features suggested that the Portage zone and surrounding Main and East Extension zones existed as a small island of maar craters of low relief in a rapidly deepening shallow basin.

This interpretation has its closest modern analogue in the Ladolam Gold Deposit, Lihir Island, Papua New Guinea. Mineralization at Lihir is believed to be less than 500,000 years old and is telescoped upon an earlier porphyry environment (Carman, 2003). Deposition of gold is still an active process at Ladolam as the hydrothermal system remains active. Host rocks at Ladolam can be divided into three groups (Carman, 2003):

- Mafic lavas composed of alkali basalt, porphyritic trachybasalt, trachyandesite, and rare trachyte and phonolite.
- Alkali intrusions that are composed of multi-phase porphyry stocks with the most voluminous phase being biotite monzonite.
- Ladolam Breccia Complex that is composed of porphyry breccias and volcanic breccias.

Porphyry breccias are dominantly monzonite composition and occur as poorly sorted, massive, matrix supported breccias with some rounding of clasts caused by magmatic milling; the clasts are supported by a cement of altered rock flour and anhydrite. The volcanic breccias are massive, moderately to poorly sorted, rock flour matrix supported breccias containing mafic clasts.

Mineralization/alteration at Ladolam can also be sub-divided into three broad phases:

- Biotite-orthoclase-anhydrite ± magnetite with minor copper-gold-molybdenum disseminated porphyry mineralization and veinlets.
- Refractory sulphide-gold mineralization associated with pervasive adularia-pyrite (leucoxene-illite) alteration near surface that comprises the bulk of the near surface bulk mineable mineralized material.
- Quartz-calcite-adularia-pyrite-marcasite ±electrum stockwork veins.

If the Ladolam Gold deposit is accepted as a reasonable genetic analogue to the Springpole deposit, then the following genetic model can be applied. This model is adapted from Barron's thesis (1996), Zabev's genetic summary (2004), and the genetic model of Armstrong et al…, (2006), as well as observations made during the 2009 through 2012 diamond drilling programs.

8.1.1 Springpole Genetic Model

The following list summarizes the genetic model of the Springpole Gold Project area.

- Intrusion into the lower crust of parental alkaline primitive and anhydrous magma slightly enriched in incompatible elements including fluorine.
- Fractionation at depth, precipitation of hornblende and apatite as early crystalline phases; the magma becomes increasingly anhydrous. Gold is retained in the melt.
- Diapiric uprise from 4 to 8 km levels into hydrous wall rock with the apex of the magma chamber at <2 km depth. Continued fractionation producing an increasingly fluorine-rich melt; feldspar of extreme composition is precipitated, and the lowered solidus allows emplacement of porphyry dikes and sills to very high crustal levels.
- High diffusivities and convection promotes water partitioning from wall rock into magma.
- The magma is quickly saturated, and the sudden pressure is released (possibly from venting) prompting the immiscible separation of fluorine and carbon dioxide-rich phases, which escapes

to high structural levels. Breccia pipes with rock fluorite and rounded clasts indicating turbulent fluidized and erosional vertical emplacement.

- Fluid pressures generate dike offshoots.
- Fluorine escapes from brecciated wall-rock causing biotization or fluoritization of breccia and wall rock. Ultimately, the fluorine-water-carbon dioxide vapours condense, resulting in the precipitation of fluorite and calcite. Magmatic gold-rich fluids permeate the breccia and surrounding porphyry, depositing porphyry style, disseminated, pyritic mineralization. The fractures along the margins of breccia pipes acts as preferred sites for later deposition of quartz, electrum, and tellurides.
- Intrusion of a series of lamprophyre and carbonatite dikes, sills, and veinlets—due to the intensity of deformation.
- The complex is then buried by conglomerates derived from the complex and other areas (Devaney, 2001b).
- Continued intense deformation and associated metamorphism manifesting as folding, strike-slip faulting and shearing, coupled with regional green schist metamorphism of the region obscures primary textures and likely leads to some (possibly minor) degree of precious metal remobilization.

9 EXPLORATION

During the winter of 2019 - 2020, First Mining initiated a program of core re-sampling in order to quantify the sulphur content of the in-pit material. A total of 8,358 samples were collected for total sulphur assays, along with 611 samples collected for bulk density determination.

Several field programs were completed by First Mining throughout 2020, with the primary purpose of collecting additional data to advance the metallurgical, geotechnical, hydrogeological, and environmental studies at Springpole through PFS level and beyond. Diamond drilling was undertaken to collect samples for metallurgical and geotechnical test work and this drilling is summarized in Section 10.

The geotechnical program targeting the pit wall area consisted of drilling and logging of inclined HQ size boreholes, packer tests, fracture surveys using acoustic televiewer, rock testing (point load tests and Brazilian tests), and multi-level piezometer installation.

In addition, a detailed geotechnical field testing and sampling program was completed over the areas of proposed mine infrastructure, which included test pit excavations (for overburden investigation), hand auguring, NQ-size borehole drilling, and ground penetrating radar (GPR) surveys in selected locations.

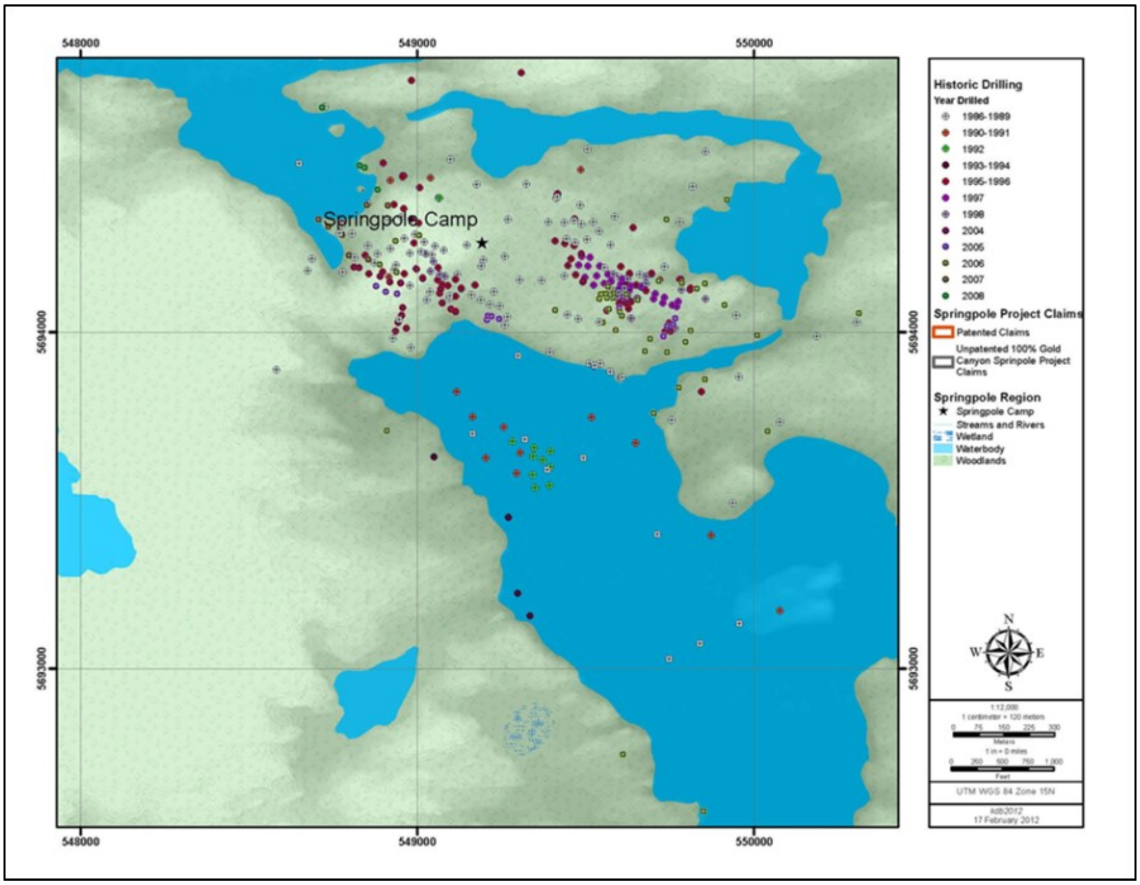
A program of condemnation drilling targeting key infrastructure areas was also commenced in 2020 and is scheduled for completion in 2021. Additional mapping and sampling of nearby trachyte outcrops was completed during the summer months and further exploration on these areas and other potential targets outside of the main resource area will continue in 2021.

10 DRILLING

10.1 Gold Canyon Drilling

During the winters of 2007 and 2008, Gold Canyon conducted diamond drill programs that completed 18 holes totalling 4,574 m (Figure 10-1). The details of the exploration work carried out are covered in Gold Canyon's internal Winter Drilling Report 2006 - 2007 (Smith, 2008a) and Winter Drilling Report 2008 (Smith, 2008b).

Figure 10-1: Springpole Gold Project Historical 2007 and 2008 Drill Hole Collar Location Map



Source: Gold Canyon, 2011

10.2 2007 Diamond Drilling Program

During the winter of 2007, Gold Canyon conducted an 11-diamond drill hole program that totalled 2,122 m of drilling. Table 10-1 summarizes drill hole collar information and significant results of the 2007 diamond drill program are summarized in Table 10-2.

Table 10-1: Summary of 2007 Winter Diamond Drilling Program

Hole ID	Azimuth (°)	Dip (°)	Length (m)	Easting (UTM)	Northing (UTM)	Elevation (m)
BL-07-374	180	-45	200	549,157	5,692,502	394
BL-07-375	180	-45	200	549,212	5,692,552	394
BL-07-376	180	-45	113	549,414	5,692,412	402
BL-07-377	180	-45	194.4	549,640	5,692,628	392.49
BL-07-378	230	-45	149	548,855	5,694,217	399
BL-07-379	230	-45	200	548,797	5,694,228	401
BL-07-380	230	-45	196.2	548,776	5,694,290	407
BL-07-381	230	-45	194	548,735	5,694,314	404
BL-07-382	240	-45	251	548,707	5,694,336	405
BL-07-383	240	-45	203	548,850	5,694,378	410
BL-07-384	230	-45	221	548,912	5,694,377	410
Total			**2,121.6**			

Notes: Coordinates in Universal Transverse Mercator (UTM); World Geodetic System 1984 (WGS84) converted from NAD27. Elevations from LiDAR.

Table 10-2: Significant Drill Intercepts from 2007 Drilling Program

Hole ID	From (m)	To (m)	Interval (m)	Au (g/t)
BL-07-374	93.93	95	1.07	0.41
	163	167	4	0.55
BL-07-375	110.55	111.24	0.69	2.32
BL-07-376	29.2	29.93	0.73	2.44
BL-07-377	105.45	105.95	0.5	3.16
	148.12	152	3.88	1.46
BL-07-378	89.62	90.16	0.54	19.32
	114.22	116	1.78	3.19
BL-07-379	56.89	57.26	0.37	14.07
	60.81	61.1	0.29	5.65
	107	107.51	0.51	2.21
	117.26	117.76	0.5	0.141
BL-07-380	116.05	116.61	0.56	1.05
	138	138.42	0.42	4.19
BL-07-383	42	47.26	5.26	9.79
BL-07-384	80.54	81.54	1	1.03
	149.36	149.91	0.55	2.85

10.3 Winter 2008 Drill Program

The winter 2008 program comprised seven core holes totalling 2,452 m and was designed to focus on step-out drilling to test the strike and down-dip potential of the new sedimentary-hosted, semi-massive sulphide environment. The first 1 km of strike potential for the sedimentary-hosted semi-massive sulphide environment has now been tested at a vertical depth of between 100 and 200 m. The results of the 2008 drilling program were inconclusive and did not return any gold intersections comparable to BL-07-383. The sedimentary hosted gold target horizon is believed to continue for at least seven additional kilometres beyond the area tested.

Table 10-3 summarizes the 2008 drilling program and Table 10-4 summarizes the significant intersections from the drilling campaign.

Table 10-3: Summary of 2008 Winter Diamond Drilling Program

Hole ID	Azimuth (°)	Dip (°)	Length (m)	Easting (UTM)	Northing (UTM)	Elevation (m)
BL08-385	240	-45	208	548,881	5,694,420	395
BL08-386	215	-45	272	548,843	5,694,489	399
BL08-387	215	-45	395	548,828	5,694,495	399
BL08-388	215	-60	356	548,828	5,694,495	399
BL08-389	258	-45	356	548,828	5,694,495	399
BL08-390	268	-45	446	548,828	5,694,495	399
BL08-391	240	-45	419	548,717	5,694,668	393

Notes: Coordinates in Universal Transverse Mercator (UTM); World Geodetic System 1984 (WGS84) converted from NAD27. Elevations from LiDAR.

Table 10-4: Significant Drill Intercepts from 2008 Drilling Program

Hole ID	From (m)	To (m)	Interval (m)	Au (g/t)
BL08-385	74.00	75.56	1.56	3.28
	167.39	168.39	1.00	2.37
BL08-386	99.28	100.25	0.97	2.61
	222.38	223.18	0.80	13.17
BL08-387	193.06	194.00	0.94	1.69
	292.71	296.67	3.96	1.63
BL08-389	167.00	168.23	1.23	2.04
	207.00	207.93	0.93	1.91
	305.92	307.59	1.67	1.47
	345.50	346.52	1.02	5.86

10.4 2010 Drill Program

Winter 2010 drilling operations began on February 17, 2010 with mobilization of two Longyear 38 drills from Boart-Longyear International's (BLI) base in Red Lake. Drilling commenced on February 23, 2010. A total of six diamond drill holes (SP10-001 through SP10-006) were drilled for a total of 1,774.5 m of HQ drilling (Figure 10-2). A summary of the 2010 drilling and significant gold and silver assays can be found in Table 10-5 and Table 10-6.

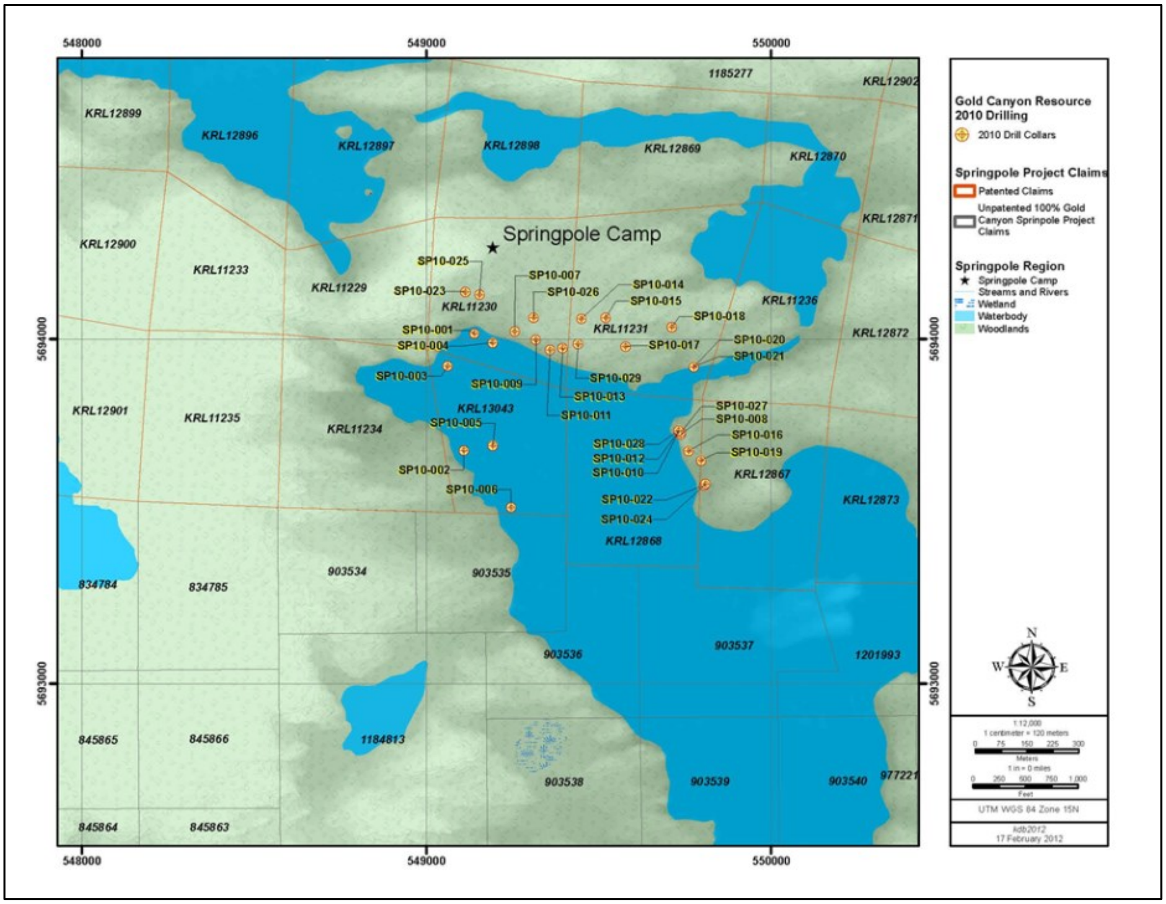
BLI pulled out of the drill program and demobilized the drills on March 10, 2010, citing critical ice thickness problems with the access ice road to Springpole camp from the South Bay Mine landing. In doing so, BLI failed to complete drill holes SP10-005 and SP10-006, and both holes ended in altered and mineralized rock. Locations of the holes from the 2010 drill program are shown on Figure 10-2.

Significant intercepts of the 2010 drill program are listed in Table 10-5.

Drilling was suspended during the ice break-up on Springpole Lake and Birch Lake as the Project has no land access route. Rodren Drilling Ltd of Winnipeg, Manitoba, was awarded the drilling contract in spring 2010 and mobilization of two Boyles 37 drills to the Project site by helicopter began in June

2010. Drilling commenced on July 5, 2010 and ended on October 17, 2010. An additional 23 diamond drill holes (SP10-007 through SP10-029) were drilled for a total of 8,662 m during this period.

Figure 10-2: Springpole Gold Project – 2010 Drill Hole Collar Location Map



Source: Gold Canyon, 2011

Table 10-5: Summary of 2010 Diamond Drilling Program

Hole ID	Azimuth (°)	Dip (°)	Length (m)	Easting (UTM)	Northing (UTM)	Elevation (m)
SP10-001	220	-45	252	549,140	5,694,017	391
SP10-002	40	-45	392	549,109	5,693,677	398
SP10-003	40	-45	225	549,062	5,693,922	391
SP10-004	220	-45	274.5	549,192	5,693,990	391
SP10-005	40	-59	268	549,193	5,693,691	392
SP10-006	40	-45	363	549,246	5,693,512	392
SP10-007	220	-45	250	549,256	5,694,022	391
SP10-008	231	-45	451	549,739	5,693,725	391
SP10-009	220	-45	322	549,318	5,693,998	391
SP10-010	242	-45	317	549,733	5,693,733	391
SP10-011	220	-45	328	549,359	5,693,969	391
SP10-012	226	-45	431	549,732	5,693,734	391
SP10-013	54	-45	313	549,396	5,693,974	391
SP10-014	36	-45	262	549,450	5,694,059	391
SP10-015	40	-45	272	549,521	5,694,062	391
SP10-016	225	-45	511	549,762	5,693,676	391
SP10-017	35	-45	298	549,578	5,693,979	391
SP10-018	38	-50	226	549,713	5,694,035	391
SP10-019	220	-45	490	549,797	5,693,648	391
SP10-020	35	-45	349	549,777	5,693,920	394
SP10-021	220	-45	502.2	549,777	5,693,920	394
SP10-022	220	-45	396	549,807	5,693,576	391
SP10-023	220	-45	454	549,113	5,694,136	391
SP10-024	220	-45	505	549,811	5,693,580	391
SP10-025	220	-45	430	549,154	5,694,129	391
SP10-026	220	-45	466	549,312	5,694,063	391
SP10-027	240	-45	115	549,739	5,693,730	391
SP10-028	245	-45	475	549,732	5,693,735	391
SP10-029	222	-45	499	549,440	5,693,986	391

Notes: Coordinates in Universal Transverse Mercator (UTM); World Geodetic System 1984 (WGS84) converted from NAD27. Elevations from LiDAR.

Table 10-6: Significant Intercepts from 2010 Drilling Program

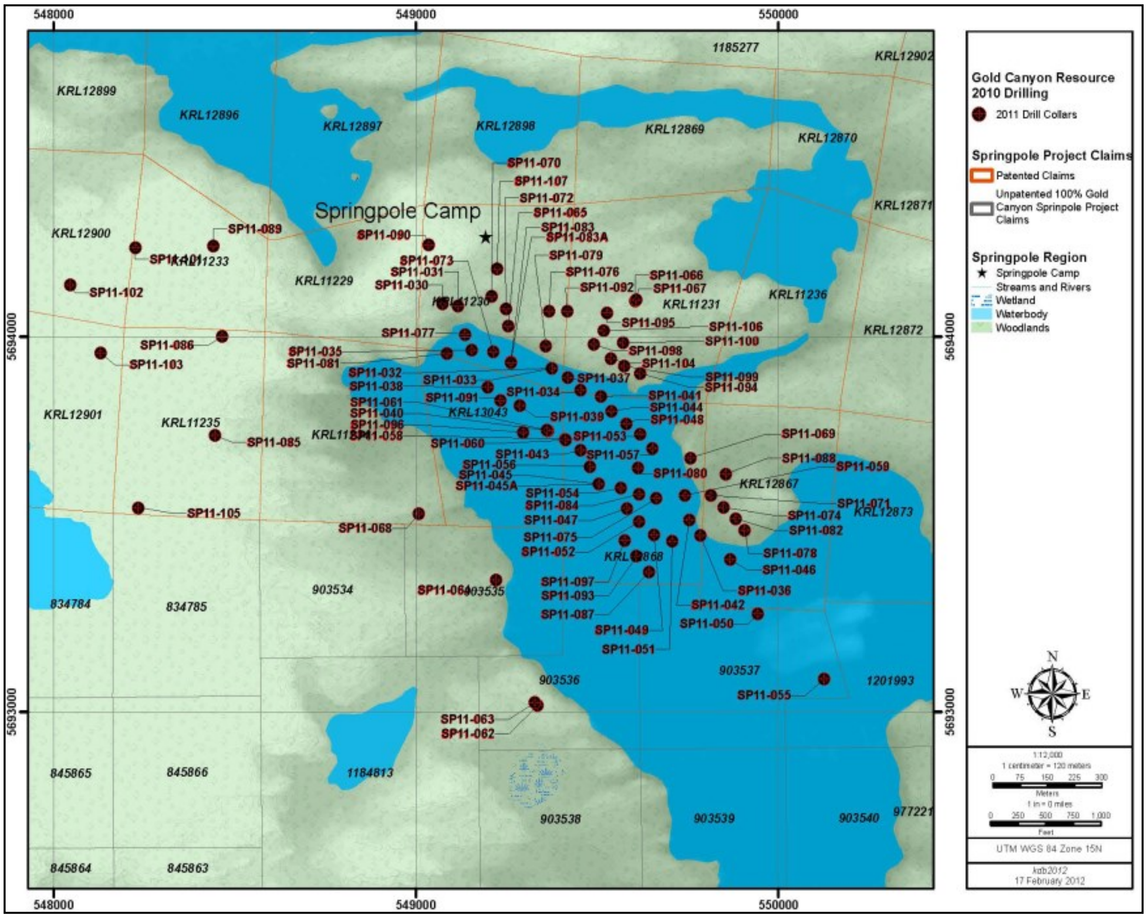
Hole ID	From (m)	To (m)	Interval (m)	Au (g/t)
SP10-001	12.5	64	51.5	0.93
SP10-002	242	335	93	2.4
SP10-004	31	182	151	0.72
SP10-006	278	363	85	0.93
SP10-007	33	250	217	1.57
SP10-008	257	451	194	1.22
SP10-009	3	167	164	1.02
SP10-011	229	323	94	2.51
SP10-012	275	408	133	0.79
SP10-016	206	511	305	1.03
SP10-019	182	489	307	1.44
SP10-024	166	391	225	1.48
SP10-026	54	407	353	1.17

10.5 2011 Drill Program

The 2011 drill program totalled 29,787 m in 82 diamond core holes. The drill hole dataset is illustrated in Figure 10-3 and summarized in Table 10-7. Five of the diamond core holes were drilled for the purpose of metallurgical testing. All of these holes (SP11-061, -065, -066, -069 and -090) were twins of previously drilled holes. The core obtained from SP11-061, -065 and -069 was not sampled in order to send the whole core for metallurgical testing. The drill core from SP11-066 and -090 was quartered and one-quarter was sent to SGS's Red Lake laboratory for assaying. The remaining three-quarters were sent to SGS's Lakefield metallurgical laboratory facility along with the whole cores.

Table 10-8 summarizes the significant gold and silver intercepts from the 2011 diamond core drilling program.

Figure 10-3: Springpole Gold Project – 2011 Drill Hole Collar Location Map



Source:Gold Canyon, 2011

Table 10-7: Summary of 2011 Diamond Drill Program

Hole ID	Azimuth (°)	Dip (°)	Length (m)	Easting (UTM)	Northing (UTM)	Elevation (m)
SP11-030	220	-45	238	549,074	5,694,088	391
SP11-031	220	-45	241	549,116	5,694,081	391
SP11-032	220	-45	70	549,376	5,693,915	391
SP11-033	220	-45	350.7	549,376	5,693,915	391
SP11-034	220	-55	379.5	549,454	5,693,857	391
SP11-035	0	-90	200.5	549,154	5,693,964	391
SP11-036	220	-45	394.5	549,785	5,693,470	391
SP11-037	220	-45	369.3	549,420	5,693,891	391
SP11-038	0	-90	202	549,199	5,693,865	391
SP11-039	220	-90	176	549,287	5,693,816	391
SP11-040	0	-90	151.5	549,364	5,693,749	391
SP11-041	220	-45	246	549,510	5,693,841	391

Hole ID	Azimuth (°)	Dip (°)	Length (m)	Easting (UTM)	Northing (UTM)	Elevation (m)
SP11-042	220	-45	411	549,755	5,693,511	391
SP11-043	0	-90	153	549,455	5,693,698	391
SP11-044	220	-45	351	549,540	5,693,802	391
SP11-045	0	-90	90	549,505	5,693,607	391
SP11-045A	0	-90	213	549,505	5,693,607	391
SP11-046	220	-45	395	549,867	5,693,406	391
SP11-047	0	-90	177	549,582	5,693,542	391
SP11-048	220	-45	360	549,581	5,693,768	391
SP11-049	0	-90	152	549,657	5,693,471	391
SP11-050	220	-45	402	549,944	5,693,262	391
SP11-051	0	-90	164	549,707	5,693,455	391
SP11-052	0	-90	158	549,616	5,693,508	391
SP11-053	220	-45	351	549,619	5,693,741	391
SP11-054	0	-90	165	549,565	5,693,597	391
SP11-055	220	-45	407.5	550,126	5,693,088	391
SP11-056	0	-90	228	549,481	5,693,653	391
SP11-057	220	-45	348	549,653	5,693,702	391
SP11-058	0	-90	159	549,411	5,693,727	391
SP11-059	220	-45	364.5	549,743	5,693,577	391
SP11-060	0	-90	255	549,413	5,693,725	391
SP11-061	0	-90	132	549,361	5,693,751	391
SP11-062	40	-45	456	549,335	5,693,018	398
SP11-063	40	-45	975	549,337	5,693,018	402.96
SP11-064	40	-45	899	549,221	5,693,351	398
SP11-065	220	-45	387.5	549,255	5,694,028	391
SP11-066	20	-45	301	549,606	5,694,095	394
SP11-067	40	-45	337	549,608	5,694,098	395
SP11-068	40	-50	902	549,008	5,693,526	398.78
SP11-069	225	-45	423	549,758	5,693,677	391
SP11-070	220	-55	491	549,209	5,694,107	391
SP11-071	220	-60	494	549,814	5,693,577	391
SP11-072	220	-55	492	549,248	5,694,073	391
SP11-073	0	-90	398	549,214	5,693,960	391
SP11-074	220	-45	498	549,849	5,693,546	391
SP11-075	0	-90	399	549,664	5,693,569	391
SP11-076	220	-45	404.5	549,368	5,694,068	391
SP11-077	0	-90	342	549,135	5,694,006	391
SP11-078	220	-45	494	549,908	5,693,485	391
SP11-079	220	-60	425	549,359	5,693,976	391
SP11-080	0	-90	420	549,614	5,693,651	391
SP11-081	0	-90	361	549,086	5,693,954	391
SP11-082	220	-45	478	549,883	5,693,515	391
SP11-083A	0	-90	144	549,262	5,693,932	391
SP11-083	0	-90	381	549,262	5,693,930	391

Hole ID	Azimuth (°)	Dip (°)	Length (m)	Easting (UTM)	Northing (UTM)	Elevation (m)
SP11-084	0	-90	349.5	549,616	5,693,580	391
SP11-085	220	-45	302	548,446	5,693,737	411
SP11-086	220	-45	302	548,465	5,694,001	408
SP11-087	0	-90	396	549,643	5,693,373	391
SP11-088	220	-60	598	549,856	5,693,634	391
SP11-089	220	-45	302	548,441	5,694,242	415.72
SP11-090	200	-45	206	549,035	5,694,244	399
SP11-091	0	-90	400.5	549,234	5,693,830	391
SP11-092	220	-55	424	549,418	5,694,068	391
SP11-093	0	-90	316.5	549,609	5,693,415	391
SP11-094	222	-50	570	549,619	5,693,902	391
SP11-095	220	-45	441	549,528	5,694,063	391
SP11-096	0	-90	327	549,295	5,693,745	391
SP11-097	0	-90	291	549,576	5,693,457	392
SP11-098	223	-45	401.5	549,491	5,693,979	391
SP11-099	223	-45	463	549,575	5,693,921	391
SP11-100	223	-50	521.5	549,572	5,693,984	391
SP11-101	220	-45	302	548,226	5,694,236	417.27
SP11-102	220	-45	302	548,047	5,694,138	419
SP11-103	220	-45	290	548,130	5,693,957	415
SP11-104	220	-45	458	549,538	5,693,942	391
SP11-105	220	-45	302	548,233	5,693,544	414
SP11-106	220	-45	509.5	549,519	5,694,016	391
SP11-107	220	-45	515	549,224	5,694,180	391
SP11-108	219	-45	540	549,483	5,694,037	391
SP11-109	224	-45	600	549,615	5,694,037	391

Notes: Coordinates in Universal Transverse Mercator (UTM); World Geodetic System 1984 (WGS84). Elevations from LiDAR.

Table 10-8: Significant Intercepts from 2011 Drilling Program

Hole ID	From (m)	To (m)	Interval (m)	Au (g/t)	Ag (g/t)
SP11-030	14	73	59	2.54	2.02
SP11-033	13	315	302	1.38	7.17
SP11-034	37	110.5	73.5	1.18	6.18
	162	331	169	1.08	6.29
SP11-035	37	68	31	1.02	3.63
	105	200.5	95.5	1.22	3.26
SP11-036	204	394.5	190.5	0.9	3.97
SP11-037	54	316.5	262.5	0.92	4.67
SP11-038	61	79	18	0.89	4.62
SP11-039	60	117	57	0.41	3.11
	132	165	33	0.53	4.48
SP11-040	51	151.5	100.5	7.59	8.82
SP11-041	161	237	76	1.5	5.6

Hole ID	From (m)	To (m)	Interval (m)	Au (g/t)	Ag (g/t)
SP11-042	9	411	402	0.76	2.86
SP11-043	42	153	111	2.03	7.01
SP11-044	132	351	219	0.71	11.81
SP11-045	36	90	54	2.15	19.13
SP11-045A	63	213	150	2.59	12.40
SP11-046	34	63	29	0.57	5.49
	238	306.5	68.5	0.82	6.74
SP11-047	22.7	177	154.3	0.99	8.69
SP11-048	121	315	194	1.11	13.79
SP11-049	20	152	132	1.37	7.60
SP11-050	139	247	108	0.54	3.3
	304	328	24	0.63	3.96
SP11-051	14	164	150	1.14	3.80
SP11-052	19	158	139	1.04	10.83
SP11-053	11.4	21	9.6	2.95	13.32
SP11-054	23	165	142	0.81	17.63
SP11-055	18	33	15	0.36	3.08
SP11-056	55.5	228	172.5	0.93	21.38
SP11-057	91.5	312	220.5	0.84	4.91
SP11-058	48.4	159	110.6	2.48	4.57
SP11-059	72	364.5	292.5	1.13	4.15
SP11-060	51	255	204	1.15	4.89
SP11-066	16	40	24	17.48	3.20
SP11-067	15	54	39	2.93	1.04
SP11-070	93	401	308	1.29	1.40
SP11-071	149	435	286	1.06	7.73
SP11-072	63	382	319	0.97	2.52
SP11-073	17	267	250	1.46	3.00
SP11-074	121	490	369	0.91	5.58
SP11-075	113	319	206	0.91	2.85
SP11-076	28	149	121	0.71	1.53
	295	387	92	0.6	2.15
SP11-077	10	87	77	0.73	0.51
	130	236	106	3.35	2.15
SP11-078	249	363	114	0.58	4.09
SP11-079	3	177.5	174.5	0.56	1.97
	312	416	104	0.59	2.13
SP11-080	48	124	76	0.62	1.9
SP11-081	92	321	229	0.82	2.42
SP11-082	85	171	86	1.07	17.95
	262	403	141	0.72	5.93
SP11-083	24	155	131	0.77	3.12

Hole ID	From (m)	To (m)	Interval (m)	Au (g/t)	Ag (g/t)
SP11-084	15	349.5	334.5	0.81	5.14
SP11-087	159	353	194	0.96	5.97
SP11-088	7	36	29	0.62	1.20
	300	346	46	0.58	6.69
	364	441	77	0.72	4.74
SP11-091	66	376	310	1.89	6.60
SP11-092	109	177	68	0.58	1.00
SP11-093	122	316.5	194.5	0.85	3.73
SP11-094	312.5	455	142.5	0.71	4.70
SP11-096	66	323	257	1.48	5.82
SP11-097	27	60	33	0.71	0.78
	200	291	91	0.79	4.67
SP11-098	3	124	121	1.68	3.67
	311.5	401.5	90	2	7.17
SP11-099	254	430	176	0.8	7.63
SP11-100	404.5	482	77.5	0.62	5.38
SP11-104	279	427	148	1.66	6.11
SP11-106	256	269	13	0.77	2.85
	344.5	472	127.5	3.52	10.72
SP11-107	247	377	130	0.73	2.40
SP11-108	381.0	446.0	65.0	3.14	11.02
SP11-109	509.5	540.0	30.5	1.09	4.52

10.6 2012 Drill Program

The 2012 drill program commenced on January 18, 2012, using the two Boyles 37 drill rigs from Rodren and one discovery EF-50 drill from the 2011 program. Three Discovery LF-75 drills, mobilized to the Springpole Gold Project via the winter road, were also used. The drill program began infilling the Portage zone based upon results of the 2011 drill program. The goal was to infill areas where inferred mineral resources had been defined in the February 2012 Mineral Resource update and to potentially expand the mineralization to the southeast.

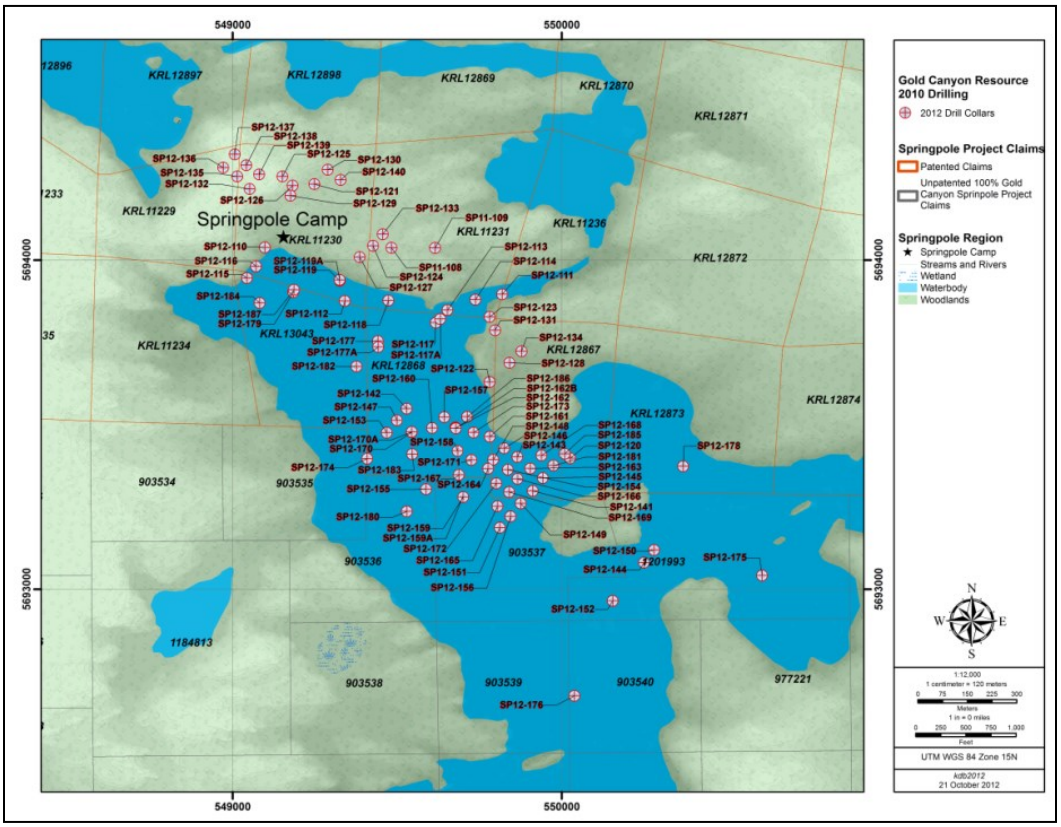
The 2012 drill program totalled 39,392 m in 98 diamond core holes. The majority of this drilling was infill drilling, but 8 holes (SG12-188 through SG12-193A) totalling 1,291 m were drilled for geotechnical purposes, and four holes totalling 1,147 m were condemnation holes drilled to test for possible mineralization in the vicinity of potential future infrastructure locations.

Condemnation holes SC12-194, SC12-196 and SC12-198 were drilled on the east side of Johnson Island and no significant assay intervals were intersected in these three drill holes. Hole SC12-195, located to the west of Johnson Island, intersected all alkaline rocks, and intersected two mineralized intervals: 151 m to 161 m (10 m) with an average grade of 0.79 g/t Au, and 169 m to 175 m (6 m) with an average grade of 1.08 g/t Au.

The objective of the geotechnical holes was to test the lake bottom conditions and the gross rock character/strength in six areas that were proposed for cofferdam locations. The suffix A in the hole ID indicates that the initial hole was lost because of poor recovery. None of these holes had any significant gold mineralization, with the exception of SG12-193A which intercepted a 28 m thick interval of low grade gold mineralization averaging 0.39 g/t Au (from 168 to 196 m).

The drill hole dataset is illustrated in Figure 10-4 and summarized in Table 10-9. Significant drill intersections from the 2012 drilling program are summarized in Table 10-10.

Figure 10-4: Springpole Gold Project – 2012 Drill Hole Collar Location Map



Source:Gold Canyon, 2012

Table 10-9: Summary of 2012 Diamond Drilling Program

Hole ID	Azimuth (°)	Dip (°)	Length (m)	Easting (UTM)	Northing (UTM)	Elevation (m)
SP12-110	0	-90	480.5	549,098	5,694,038	391
SP12-111	220	-45	568	549,819	5,693,896	394
SP12-112	0	-90	407	549,341	5,693,876	391
SP12-113	221	-45	496	549,653	5,693,848	391
SP12-114	220	-45	569.6	549,738	5,693,880	395
SP12-115	0	-90	527	549,044	5,693,945	391
SP12-116	0	-90	449	549,071	5,693,981	391
SP12-117	220	-45	75.2	549,618	5,693,809	391
SP12-117A	220	-45	426	549,631	5,693,821	392
SP12-118	220	-45	413	549,474	5,693,877	391
SP12-119	0	-90	22.6	549,326	5,693,937	391
SP12-119A	0	-90	449	549,325	5,693,940	391
SP12-120	220	-45	332	550,026	5,693,397	394
SP12-121	220	-45	518	549,249	5,694,231	397
SP12-122	220	-45	587	549,781	5,693,629	391
SP12-123	221	-45	566	549,781	5,693,827	392
SP12-124	220	-45	491.5	549,427	5,694,043	391
SP12-125	221	-45	392	549,152	5,694,254	396
SP12-126	219	-45	509	549,183	5,694,226	392
SP12-127	221	-45	547	549,386	5,694,009	391
SP12-128	222	-45	654	549,841	5,693,688	393
SP12-129	221	-45	494	549,176	5,694,196	392
SP12-130	219	-45	614	549,289	5,694,275	400
SP12-131	222	-45	656	549,798	5,693,787	393
SP12-132	220	-45	287	549,052	5,694,216	393
SP12-133	220	-45	527	549,456	5,694,078	391
SP12-134	220	-45	701	549,878	5,693,723	397
SP12-135	220	-45	305	549,014	5,694,254	399

Hole ID	Azimuth (°)	Dip (°)	Length (m)	Easting (UTM)	Northing (UTM)	Elevation (m)
SP12-136	220	-45	251	548,972	5,694,281	402
SP12-137	220	-45	377	549,007	5,694,321	401
SP12-138	220	-45	404	549,046	5,694,291	400
SP12-139	220	-45	341	549,081	5,694,260	393
SP12-140	212	-55	618.5	549,328	5,694,243	401
SP12-141	0	-90	516	549,912	5,693,299	391
SP12-142	0	-90	361.5	549,529	5,693,549	391
SP12-143	0	-90	432	549,865	5,693,402	391
SP12-144	0	-90	473	550,250	5,693,081	391
SP12-145	0	-90	478	549,943	5,693,338	391
SP12-146	0	-90	455	549,825	5,693,428	391
SP12-147	0	-90	499.5	549,500	5,693,513	391
SP12-148	0	-90	534	549,792	5,693,394	391
SP12-149	0	-90	500	549,876	5,693,260	391
SP12-150	0	-90	602	550,280	5,693,119	391
SP12-151	0	-90	503	549,812	5,693,187	391
SP12-152	0	-90	671	550,155	5,692,964	391
SP12-153	0	-90	477	549,469	5,693,475	391
SP12-154	0	-90	525	549,836	5,693,363	391
SP12-155	0	-90	443	549,588	5,693,304	391
SP12-156	0	-90	435	549,844	5,693,221	391
SP12-157	0	-90	379.5	549,643	5,693,523	391
SP12-158	0	-90	395	549,684	5,693,420	391
SP12-159	0	-90	59	549,701	5,693,280	391
SP12-159A	0	-90	355.5	549,701	5,693,280	391
SP12-160	0	-90	420	549,606	5,693,490	391
SP12-161	0	-90	362	549,781	5,693,463	391
SP12-162	0	-90	29	549,678	5,693,489	391
SP12-162B	0	-90	468	549,678	5,693,488	391
SP12-163	0	-90	431	549,904	5,693,367	391

Hole ID	Azimuth (°)	Dip (°)	Length (m)	Easting (UTM)	Northing (UTM)	Elevation (m)
SP12-164	0	-90	464	549,776	5,693,366	391
SP12-165	0	-90	495.5	549,805	5,693,252	391
SP12-166	0	-90	354	549,866	5,693,335	391
SP12-167	0	-90	400	549,688	5,693,347	391
SP12-168	0	-90	473	549,939	5,693,406	394
SP12-169	0	-90	362	549,840	5,693,295	391
SP12-170	0	-90	257	549,544	5,693,477	391
SP12-170A	0	-90	458	549,544	5,693,478	391
SP12-171	0	-90	440	549,726	5,693,393	391
SP12-172	0	-90	405.1	549,801	5,693,322	391
SP12-173	0	-90	434	549,732	5,693,476	391
SP12-174	0	-90	506	549,409	5,693,397	391
SP12-175	0	-90	384.2	550,609	5,693,042	391
SP12-176	0	-90	296	550,039	5,692,676	391
SP12-177	0	-90	30	549,442	5,693,754	391
SP12-177A	0	-90	450	549,442	5,693,754	391
SP12-178	0	-90	395	550,369	5,693,373	391
SP12-179	0	-90	381	549,186	5,693,905	391
SP12-180	0	-90	440	549,529	5,693,236	391
SP12-181	0	-90	350	549,975	5,693,376	394
SP12-182	0	-90	395	549,377	5,693,676	391
SP12-183	0	-90	449	549,546	5,693,410	391
SP12-184	0	-90	398	549,082	5,693,870	391
SP12-185	0	-90	371	550,009	5,693,411	395
SP12-186	0	-90	468	549,713	5,693,524	391
SP12-187	0	-90	394	549,186	5,693,910	391
SP12-197	0	-90	440	549,511	5,693,612	391
SP12-199	0	-90	426	549,342	5,693,784	391
SC12-194	0	-90	323	551,063	5,692,408	391.06
SC12-195	0	-90	302	549,351	5,692,297	391.25

Hole ID	Azimuth (°)	Dip (°)	Length (m)	Easting (UTM)	Northing (UTM)	Elevation (m)
SC12-196	0	-90	200	551,476	5,692,893	391
SC12-198	0	-90	321.5	551,597	5,692,712	391
SG12-188	0	-86	200	550,873	5,693,281	396
SG12-189	0	-90	221	551,348	5,693,093	391
SG12-190	0	-90	200	549,415	5,691,094	391.79
SG12-191	0	-90	200	550,117	5,692,510	391
SG12-192	0	-90	12	551,703	5,691,409	391.3
SG12-192A	0	-90	235	551,703	5,691,409	391.3
SG12-193	0	-90	19.5	549,918	5,692,523	391
SG12-193A	0	-90	203	549,918	5,692,523	391

Notes: Coordinates in Universal Transverse Mercator (UTM); World Geodetic System 1984 (WGS84). Elevations from LiDAR.

Table 10-10: Significant Intercepts from 2012 Drilling Program

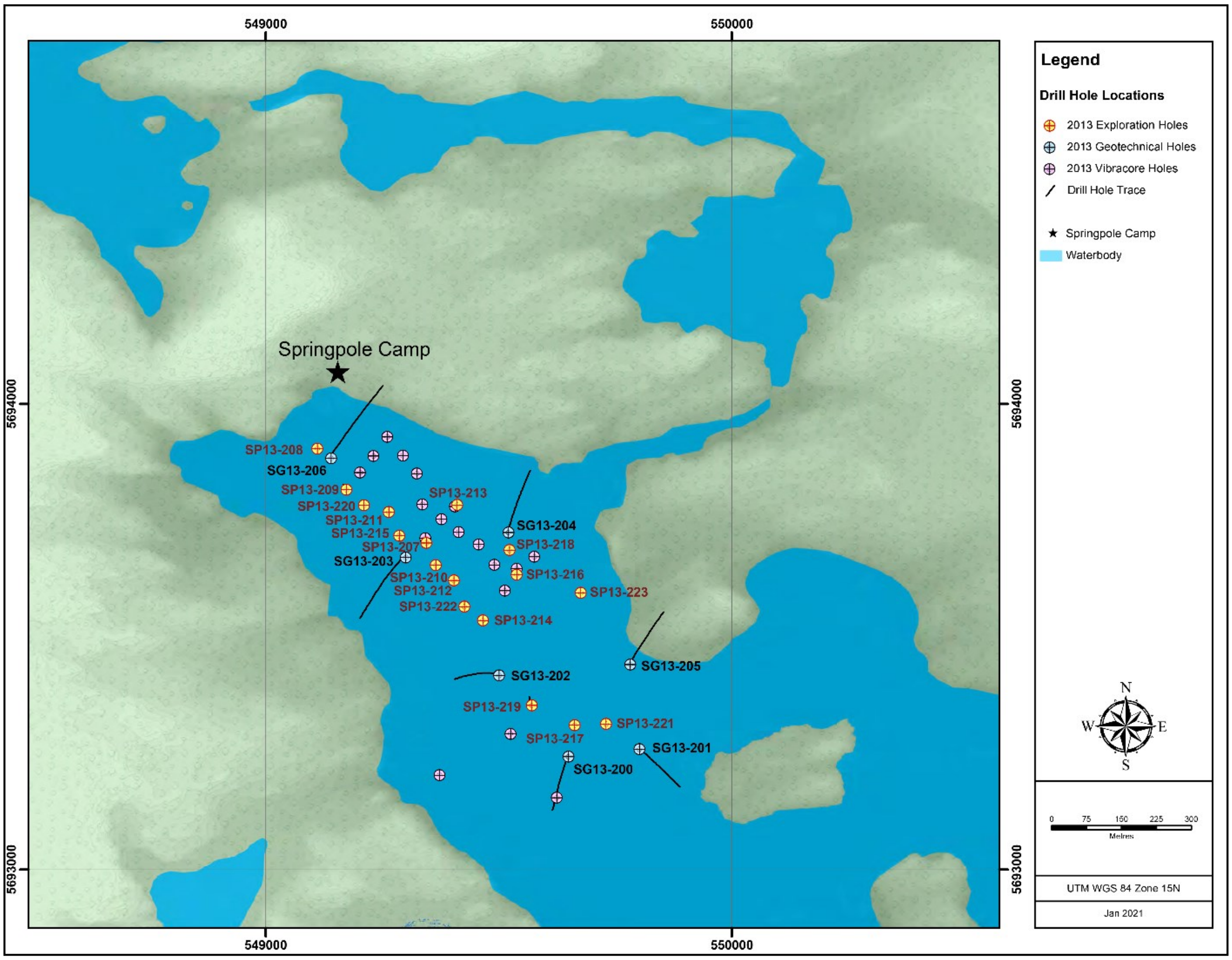
Hole ID	From (m)	To (m)	Interval (m)	Au (g/t)	Ag (g/t)
SP12-127	251.0	398.0	147.0	1.14	5.47
SP12-128	230.0	549.0	319.0	1.02	4.81
SP12-131	301.3	546.0	244.7.0	0.80	5.84
SP12-146	77.0	91.0	14.0	5.03	99.69
SP12-158	16.7	60.2	43.5	1.81	3.23
SP12-160	23.0	384.0	361.0	1.08	8.83
SP12-163	130.9	265.0	134.1	0.91	13.57
SP12-181	157.0	225.0	68.0	0.72	8.74
SP12-183	202.0	385.0	183.0	0.61	3.47
SP12-186	114.0	240.0	126.0	1.17	2.63

10.7 2013 Drill Programs

The following description of the 2013 drill programs is extracted from Muntzert (2014).

During 2013, Gold Canyon drilled 24 diamond drill holes totaling 5,394.5 m, and 18 Vibracore holes to sample the very upper portion between lake bottom and bedrock totaling 720.8 m. Drill hole locations are shown on Figure 10.5.

Figure 10-5: Springpole Gold Project – 2013 Drill Hole Collar Location Map



Source: Gold Canyon, 2013

The drilling program was conducted in three distinct phases over the year as summarized in Table 10-11.

Table 10-11: Summary of 2013 Diamond Drilling Programs

Campaign	Start Date	Finish Date	Holes	Metres
Winter	29-Jan	12-Mar	7	2,401.5
Summer	18-Jun	27-Jul	17	2,993.0
Fall	27-Sep	19-Oct	18	720.8

10.7.1 2013 Winter Drilling (Oriented Core Program):

The winter program consisted of seven inclined diamond drill holes drilled from the ice on Springpole Lake, with azimuths more or less perpendicular to the proposed pit walls.

These holes were drilled both to explore for additional mineralization outside the proposed pit wall, and to obtain further structural and geotechnical data around the proposed open-pit area.

Between January and March 2013, Gold Canyon drilled a total of 2,401.5 m in the seven holes (SG13-200 to SG13-206) as summarized in Table 10-12. Oriented core logging was completed, although final processing of the data acquired for these drillholes was not completed during 2013 as it was postponed until a future date.

Table 10-12: 2013 Oriented-Core Drilling Program

Hole ID	Easting (UTM)	Northing (UTM)	Elevation (m)	Length (m)	Azimuth (°)	Dip (°)
SG13-200	549,649	5,693,248	391	351	200	-70
SG13-201	549,801	5,693,265	391	339	135	-70
SG13-202	549,500	5,693,423	391	351	270	-70
SG13-203	549,300	5,693,676	391	349.5	225	-65
SG13-204	549,521	5,693,730	391	351	019	-65
SG13-205	549,782	5,693,447	391	349.5	030	-70
SG13-206	549,140	5,693,889	391	310.5	040	-50

Notes: Coordinates in Universal Transverse Mercator (UTM); World Geodetic System 1984 (WGS84). Elevations from LiDAR.

Three of the drill holes (SG13-201, SG13-205 and SG13-206) encountered multiple zones of mineralization as shown in Table 10-13 below.

Table 10-13: Summary of 2013 Drill Results from SG13-200 to SG13-206

Drill Hole	From (m)	To (m)	Length (m)	Au (g/t)	Ag (g/t)
SG13-200	No significant values				
SG13-201	148.0	196.0	48.0	0.66	4.76
	254.0	264.0	10	1.12	12.84
	267.0	284.0	17	0.56	7.27
SG13-202	No significant values				
SG13-203	No significant values				
SG13-204	No significant values				
SG13-205*	28.0	37.0	9.0	0.92	3.33
	99.0	103.0	4.0	2.29	110.30
	226.0	327.0	101.0	0.93	12.36
	339.0	349.5	10.5	0.64	4.80
SG13-206	68.0	91.5	23.5	1.27	4.56
	108.0	175.0	67.0	1.94	2.94
including	145.0	153.0	8.0	4.44	6.35
	241.0	255.0	14.0	1.28	2.44
	277.0	289.0	12.0	1.33	1.87

* Drill hole ended in mineralization.

10.7.2 2013 Summer Drilling

The 2013 summer drilling program was essentially an infill program designed to potentially support upgrade of Inferred Mineral Resources (located in areas with sparse drilling or along the edges of the drilling) to Indicated Mineral Resources.

In June and July 2013, 17 diamond drill holes totalling 2,993 m were drilled from barges on Springpole Lake. All of the drill holes were drilled vertically, and hole details are summarized in Table 10-14.

Table 10-14: Summary of 2013 Summer Diamond Drilling Program

Hole ID	Easting (UTM)	Northing (UTM)	Elevation (m)	Length (m)	Azimuth (°)	Dip (°)
SP13-207	549,344	5,693,707	391	224	0	-90
SP13-208	549,110	5,693,910	391	152	0	-90
SP13-209	549,173	5,693,822	391	200	0	-90
SP13-210	549,364	5,693,659	391	200	0	-90
SP13-211	549,264	5,693,774	391	158	0	-90
SP13-212	549,403	5,693,627	391	176	0	-90
SP13-213	549,410	5,693,789	391	125	0	-90
SP13-214	549,465	5,693,542	391	200	0	-90
SP13-215	549,286	5,693,723	391	179	0	-90
SP13-216	549,538	5,693,639	391	155	0	-90
SP13-217	549,663	5,693,316	391	248	0	-90
SP13-218	549,523	5,693,693	391	122	0	-90
SP13-219	549,571	5,693,358	391	176	0	-90
SP13-220	549,210	5,693,788	391	201	0	-90
SP13-221	549,730	5,693,319	391	176	0	-90
SP13-222	549,426	5,693,570	391	176	0	-90
SP13-223	549,676	5,693,601	391	125	0	-90

Notes: Coordinates in Universal Transverse Mercator (UTM); World Geodetic System 1984 (WGS84). Elevations from LiDAR

Significant intercepts from the 2013 summer drilling (holes SP13-207 to SP13-223) are shown in Table 10-15.

Table 10-15: Significant Intercepts from 2013 Summer Drilling Program

Drill Hole	From (m)	To (m)	Length (m)	Au (g/t)	Ag (g/t)
SP13-207*	70.5	89.6	19.1	0.61	4.63
	98.0	137.6	39.6	6.51	18.00
including	116.0	127.0	11.0	22.04	35.09
and including	119.0	125.0	6.0	36.31	47.20
	147.0	157.0	10.0	0.75	0.57
	182.0	224.0	42.0	3.11	24.91
Including	186.0	196.0	10.0	4.28	29.20
and including	215.0	224.0	9.0	5.54	49.56
SP13-208*	116.0	136.0	20.0	1.55	6.19
including	134.0	136.0	2.0	6.22	0.40
	144.0	152.0	8.0	0.78	8.08
SP13-209	92.0	109.0	17.0	1.47	0.52
	121.0	161.0	40.0	0.60	0.96
	171.0	175.0	4.0	1.04	1.80
	191.5	199.0	7.5	0.81	0.36
SP13-210*	103.0	109.0	6.0	1.21	1.20
	157.0	200.0	43.0	1.17	3.46
SP13-211*	73.0	158.0	85.0	2.13	4.21
including	91.0	93.0	2.0	10.91	9.60
including	143.0	150.0	7.0	4.15	9.17
SP13-212	72.0	86.0	14.0	0.79	0.95
	102.0	122.0	20.0	3.61	6.05
including	102.0	104.0	2.0	12.78	14.20
including	120.0	122.0	2.0	16.66	20.20
	160.0	172.0	12.0	0.87	1.11
SP13-213	51.0	86.0	35.0	0.77	8.68
SP13-214	81.0	87.0	6.0	0.68	3.22
	121.0	123.0	2.0	1.90	19.40
SP13-215	153.0	155.0	2.0	2.54	5.70
SP13-216	56.0	110.0	54.0	1.05	4.48
	122.0	150.0	28.0	0.87	3.04
SP13-217	140.0	160.0	20.0	0.53	2.06
	190.0	242.0	52.0	0.97	4.19
SP13-218	78.0	100.0	22.0	0.93	1.61
SP13-219	50.0	56.0	6.0	1.02	15.42
	114.0	120.0	6.0	2.08	13.67
SP13-220	58.3	107.7	49.4	0.91	1.38
SP13-221	157.0	173.0	16.0	0.47	2.51
SP13-222	No significant values				
SP13-223	No significant values				

*Drill hole ended in mineralization.

10.7.3 2013 Fall Drilling:

A review of the drill hole logs in the Portage zone that were collared on Springpole Lake revealed that the recovery in the upper parts of many holes was generally very poor. There are many instances where recovery was zero to less than 20% in the zone between the lake bottom and the top of bedrock. Core recovered from this zone contained very soft, clay-rich lake bottom sediments, till and very altered alkaline intrusive rocks which were often highly mineralized, albeit very clay altered. The thickness of this zone varied from a few metres to over 75 m.

Because of the uncertainty about the thickness and composition of the “unrecovered” zone, a new drilling technique was designed by Rodren Drilling Ltd. to try to get better recovery. The new technique employed a combination of standard soil sampling tools and sampling techniques for the very soft material and the use of Vibracore equipment to penetrate and sample the more competent sediments/rocks. Further details of the drilling procedures for these holes can be found in Muntzert, 2014.

Eighteen holes totalling 720.9 m were drilled from a barge on Springpole Lake using this new technique (Table 10-16). Good recovery of soft sediments and till was accomplished, and in three holes, bedrock was recovered. These holes established that the Portage zone is covered with up to 71 m of soft clay lake bottom sediments and till. Table 10-17 summarizes the results.

Table 10-16: Summary of 2013 Vibracore Drilling Program

Hole ID	Easting (UTM)	Northing (UTM)	Elevation (m)	Length (m)	Azimuth (°)	Dip (°)
SV13-224	549,377	5,693,753	391	45.72	0	-90
SV13-225	549,342	5,693,712	391	71.3	0	-90
SV13-226	549,405	5,693,781	391	46.6	0	-90
SV13-227	549,260	5,693,930	391	33.5	0	-90
SV13-228	549,231	5,693,889	391	40.1	0	-90
SV13-229	549,202	5,693,853	391	43.3	0	-90
SV13-230	549,294	5,693,890	391	41.2	0	-90
SV13-231	549,324	5,693,851	391	34.4	0	-90
SV13-232	549,336	5,693,785	391	56.4	0	-90
SV13-233	549,413	5,693,726	391	43.3	0	-90
SV13-234	549,456	5,693,699	391	48.8	0	-90
SV13-235	549,490	5,693,655	391	31.5	0	-90
SV13-236	549,512	5,693,600	391	38.1	0	-90
SV13-237	549,538	5,693,647	391	24.5	0	-90
SV13-238	549,576	5,693,672	391	15	0	-90
SV13-239	549,624	5,693,155	391	43.3	0	-90
SV13-240	549,525	5,693,292	391	31.3	0	-90
SV13-241	549,373	5,693,203	401	32.6	0	-90

Notes: Coordinates in Universal Transverse Mercator (UTM); World Geodetic System 1984 (WGS84). Elevations from LiDAR.

Table 10-17: Summary of Vibracore Drilling Results, 2013 Program

Sample Material	Holes Intersecting	Average Depth (m)	Depth Range (m)
Water	18	21.5	6.6 to 32.7
High Organics	14	3.7	0.0 to 4.9
Clay	17	6.0	0.0 to 10.6
Till	14	8.7	6.0 to 31.7
Bedrock	3	N/A	to 6.2

Grab samples were taken from the recovered material from this drilling and submitted to SGS Laboratories in Red Lake, Ontario. These were assayed by fire assay for gold and silver and by multi-element inductively-coupled plasma (ICP) methods.

The primary reason that this Vibracore drilling program was developed and undertaken was to explore the "zone of poor recovery" for mineralized alkaline bedrock that had high gold and silver grades. This program firmly established that the zone between lake bottom and the top of bedrock is essentially barren of any significant gold and silver mineralization.

10.7.4 2013 Geotechnical and Structural Program (SRK):

SRK were contracted by Gold Canyon to supervise the 2013 geotechnical and structural/geological program. This program was conducted at two separate times of the year, but data collected from each part was combined into one final report by C. Nagy (SRK, 2013).

During the winter drilling program, SRK geotechnical engineers were on site to train, assist and supervise the drilling crews and Gold Canyon geologists. The seven drill holes in the winter campaign were drilled using HQ3 equipment (triple tube) and Reflex ACT 2 orientation tools. Further details of the drilling procedures for these holes can be found in Muntzert (2014).

One of the objectives of this geotechnical program was to obtain data regarding the orientation (strike and dip) of faults, shear zones, joints, foliation, etc. in the rocks that were intersected in the seven drill holes. Nagy used this data to direct much of his structural geological modeling. This data defined several fault offsets of the Portage Deposit which could be used to guide additional exploration in areas where previous geological understanding was limited and poorly understood.

As part of SRK's structural work which started in September 2013, three structural geologists from SRK spent a total of 18 days at Springpole where they conducted targeted geological mapping, structural observations, and re-logging of 18 drill holes. Surface observations were then integrated with regional 2D data sets and historical geological mapping to determine the location of major structures in the Springpole area.

Observations and interpretations made in 2D were subsequently integrated in Leapfrog™ software to model structures in 3D. The 3D data included data from the Springpole drill hole database, as well as from the 18 drill holes re-logged by SRK, and from oriented core data gathered during the winter drilling program. The following drill hole data fields were used to define the modelled structures: RQD, lithology, argillic alteration, and structure (faults and shear zones). Observations from a previous structural interpretation (SRK, 2011) were also considered throughout the 3D modelling process.

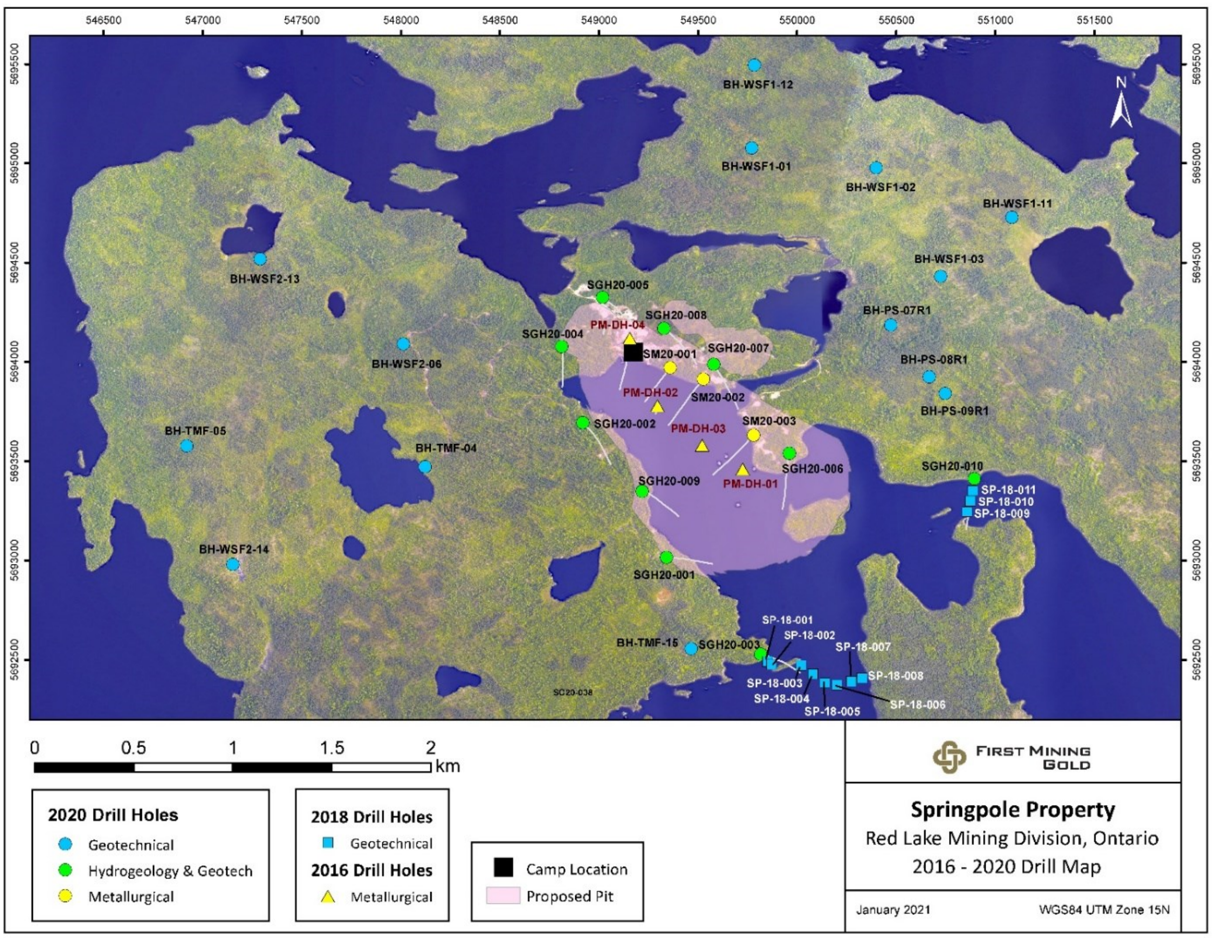
Further details on the structural modelling work undertaken by SRK can be found in the internal report by C. Nagy (SRK, 2013).

10.8 2016 Drill Program

The 2016 drill program was implemented by First Mining to collect additional material from the Portage zone so that additional metallurgical testing could be carried out. In total, 1,712 m were drilled in the four holes (PM-DH-01 to PM-DH-04).

The 2016 drill hole locations are illustrated on Figure 10-6 and hole information is summarized in Table 10-18. Significant drill intersections from the 2016 drilling program are summarized in Table 10-19.

Figure 10-6: Springpole Gold Project – 2016 to 2020 Drill Hole Collar Location Map



Source: SRK, 2021

Table 10-18: Summary of 2016 Metallurgical Drilling Program

Hole ID	Easting (UTM)	Northing (UTM)	Elevation (m)	Length (m)	Azimuth (°)	Dip (°)
PM-DH-01	549,725	5,693,460	391	450	0	-90
PM-DH-02	549,295	5,693,780	391	437	0	-90
PM-DH-03	549,520	5,693,580	391	436	0	-90
PM-DH-04	549,155	5,694,120	391	389	190	-50

Notes: Coordinates in Universal Transverse Mercator (UTM); World Geodetic System 1984 (WGS84). Elevations from LiDAR.

Table 10-19: Significant Intercepts from 2016 Metallurgical Drilling Program

Hole	From (m)	To (m)	Length (m)	Au (g/t)	Ag (g/t)
PM-DH-01	16.5	60	43.5	1.57	11.59
and	84	371	287	1.25	5.48
including	107	118	11	4.95	5.68
and including	201	236	35	2.79	13.8
PM-DH-02	74	332	258	1.87	8.24
including	171	175.5	4.5	17.97	47.57
and including	193	211	18	3.22	14.42
and including	233	237	4	14.66	53.15
PM-DH-03	50	294	244	1.2	6.96
and	312	357	45	2.7	10.42
including	332	338	6	10	26.98
and	371	399	28	0.75	4.54
PM-DH-04	7	11	4	3.73	0.86
and	16	18	2	15.66	5.1
and	22	23	1	4.95	1.32
and	88	90	2	3.91	0.35
and	104	109	5	4.01	1.05
and	134	141	7	0.59	0.73
and	146.43	154.88	8.45	0.82	0.8
and	186.32	333	146.68	2.15	4.88
including	199.15	205	5.85	11.22	4.79
and including	245	265	20	3.9	4.53

10.9 2018 Drill Program

In 2018, First Mining carried out a limited geotechnical drill program to test the integrity of ground relevant to cofferdam construction and characterize the dyke foundation materials. Eleven short holes were drilled totalling 243 m.

The drill hole locations are illustrated on Figure 10-6 above and hole information is summarized in Table 10-20 below. None of the holes intersected the mineralized domains and none have any impact on the Mineral Resource Estimates presented in Section 14 of this Report.

Table 10-20: Summary of 2018 Geotechnical Drilling Program

Hole ID	Easting (UTM)	Northing (UTM)	Elevation (m)	Length (m)	Azimuth (°)	Dip (°)
SP-18-001	549,851	5,692,492	391	13.23	0	-90
SP-18-002	549,871	5,692,478	400	9.1	0	-90
SP-18-003	550,020	5,692,471	391	12.6	0	-90
SP-18-004	550,079	5,692,429	400	48.75	0	-90
SP-18-005	550,139	5,692,384	400	30.95	0	-90
SP-18-006	550,200	5,692,373	381	14.5	0	-90
SP-18-007	550,274	5,692,392	400	28.9	0	-90
SP-18-008	550,327	5,692,409	391	7.5	0	-90
SP-18-009	550,857	5,693,246	391	18.9	0	-90
SP-18-010	550,872	5,693,300	397	39.2	0	-90
SP-18-011	550,886	5,693,353	400	19.58	0	-90

Notes: Coordinates in Universal Transverse Mercator (UTM); World Geodetic System 1984 (WGS84). Elevations from LiDAR

10.10 2020 Metallurgical Drill Program

The 2020 diamond drilling program was implemented to collect additional material for metallurgical testing within the immediate vicinity of the proposed open pit. Approximately 1,182 m of drilling was completed in three drill-holes (SM20-001 to SM20-003) as summarized in Table 10-21.

Table 10-21: 2020 Metallurgical Drill Holes

Hole ID	Easting (UTM)	Northing (UTM)	Elevation (m)	Length (m)	Azimuth (°)	Dip (°)
SM20-001	549,359	5,693,971	391	350	220	-50
SM20-002	549,526	5,693,914	391	396	220	-45
SM20-003	549,779	5,693,631	391	436.5	220	-50

Notes: Coordinates in Universal Transverse Mercator (UTM); World Geodetic System 1984 (WGS84). Elevations from LiDAR.

The 2020 metallurgical drill hole locations are illustrated on Figure 10-6 above and significant drill intersections are summarized in Table 10-22.

Table 10-22: Significant Intercepts from 2020 Metallurgical Drilling Program

Hole ID	From (m)	To (m)	Length (m)	Au (g/t)	Ag (g/t)
SM20-001	5	68	63	1	4.28
including	20	30	10	2.48	10.3
and	112	153	41	6.53	12.36
including	135	137	2	122.79	103.11
and	228	253	25	0.58	4.47
and	293	325	32	1.31	5.19
and	334	347	13	0.47	1.39
SM20-002	85	87	2	1.74	17.94
and	129	136	7	0.67	2.67
and	229	389	160	1.03	6.43
and	372	374	2	7.94	31.01
SM20-003	130	155	25	0.68	10.76
and	186	436.5	250.5	1.52	6.61
including	214	244	30	3.62	7.71
and including	277	315	38	2.37	13.92
and including	343	350	7	3.2	8.77

10.11 2020 Geotechnical Drill Program

In 2020, 24 diamond drill holes totalling 4,091 m were completed in order to obtain additional geotechnical data in both the pit wall area and the areas of planned mine infrastructure. The ten holes which targeted the pit wall were also utilized to collect hydrogeological data.

Locations of the holes are illustrated on Figure 10-6 above and hole information is summarized in Table 10-23.

Table 10-23: Summary of 2020 Geotechnical Drilling Program

Hole ID	Easting (UTM)	Northing (UTM)	Elevation (m)	Length (m)	Azimuth (°)	Dip (°)	Location
BH-PS-07R1	550,473	5,694,185	400	10.8	0	-90	Plant Site
BH-PS-08R1	550,666	5,693,924	399	11.23	0	-90	Plant Site
BH-PS-09R1	550,746	5,693,841	399	8.22	0	-90	Plant Site
BH-TMF-04	548,122	5,693,471	413.66	14.03	0	-90	TMF
BH-TMF-05	547,428	5,694,198	413.62	9.63	0	-90	TMF
BH-TMF-15	549,463	5,692,557	395.89	11.19	0	-90	TMF
BH-WSF1-01	549,772	5,695,076	398.97	8.36	0	-90	WSF
BH-WSF1-02	550,399	5,694,977	400.18	8.13	0	-90	WSF
BH-WSF1-03	550,726	5,694,431	404.85	11.12	0	-90	WSF
BH-WSF1-11	551,085	5,694,725	399.41	11.25	0	-90	WSF
BH-WSF1-12	549,785	5,695,494	395.4	8.21	0	-90	WSF
BH-WSF2-06	548,012	5,694,089	420.56	8.24	0	-90	WSF
BH-WSF2-13	547,290	5,694,515	401.89	9.57	0	-90	WSF
BH-WSF2-14	547,152	5,692,980	404.55	9.53	0	-90	WSF
SGH20-001	549,339	5,693,015	403.51	400.1	90	-55	Pit Wall
SGH20-002	548,918	5,693,693	394.92	400	137	-50	Pit Wall
SGH20-003	549,813	5,692,527	393.63	410	96.4	-50	Pit Wall
SGH20-004	548,811	5,694,076	408.58	293	169	-50	Pit Wall
SGH20-005	549,019	5,694,324	408.39	401	121	-53	Pit Wall
SGH20-006	549,959	5,693,538	399.71	440	190	-52	Pit Wall
SGH20-007	549,578	5,693,988	400.13	401	147	-52	Pit Wall
SGH20-008	549,328	5,694,167	400.81	413	125	-52	Pit Wall
SGH20-009	549,217	5,693,349	397.16	392	125	-52	Pit Wall
SGH20-010	550,894	5,693,412	393.24	401	189	-50	Pit Wall

Notes: Coordinates in Universal Transverse Mercator (UTM); World Geodetic System 1984 (WGS84). Elevations from LiDAR

10.12 Drill Collar Surveying

All historical holes drilled prior to 2010 were surveyed using various earth projections, either NAD27 (North American Datum 1927) Canada, WGS or NAD83 projections. In September 2006, W.J. Bowman Ltd. of Dryden, Ontario, surveyed 275 historic drill hole collars from collar numbers BL-1 through BL-373. For the purposes of inclusion in the data set for 3D modelling, all the historical collar locations were converted to the UTM WGS84 projection.

For the 2007 and 2008 drill programs, the drill hole collars were located and surveyed using a handheld GPS and recorded in UTM NAD27 Canada projection. All the collar survey information was converted to WGS84 and field checked against collar locations using handheld Trimble GeoXH DGPS.

The 2010 to 2012 drill hole collars were initially surveyed using handheld GPS devices. During the initial phases of the offshore 2010 drill program, drill hole collars on the lake ice were surveyed by handheld, real-time differential GPS with an average accuracy of 4 to 5 m and recorded in UTM NAD27 Canada projection. On-shore drill holes were initially located with handheld GPS and once the drill hole was complete, the hole location was temporarily marked; subsequently, the collars were surveyed using a Trimble GeoXH handheld DGPS device with an external antenna giving sub-metre (~10 cm) location accuracy.

For the offshore drill programs (2011 to 2013), drills were mounted on barges, the drill sites were marked by floating buoy and located using the Trimble GeoXH from a boat. All onshore drill collars were located and subsequently surveyed using the Trimble GeoXH. At the beginning of the winter 2011 drill program, the UTM WGS84 projection was adopted as the standard for surveying drill collars and other surface landmarks. All previously recorded UTM measurements were converted accordingly.

All drill site locations for inclined drill holes, onshore or offshore on the ice, were marked using two to four painted laths aligned along strike either side of the proposed drill hole location. These laths were used as fore- and back-sights for setting the drill location and orientation. Inclination of the drill hole was checked on the drill head, prior to commencing drilling, using either a Brunton compass or inclinometer accurate to half of one degree.

All drill holes completed by First Mining from 2016 to 2020 have been surveyed using the UTM WGS84 projection.

10.13 Oriented Core Surveying

Oriented core measurements were collected from a total of 44 drill holes (added to which are the six holes drilled in 2013). Oriented core is used to evaluate the structural geology by allowing the geologists to measure the real angular relationships, as opposed to apparent angles. The tool used was the ACT 2 from Reflex Technologies. This system is fully digital, using solid-state 3-axis accelerometers to record the orientation of the core-barrel when the core is taken off-bottom at the end of each drill run. There were significant problems encountered during the winter 2011 drill program due to tool failures. Some oriented core information was collected, but too little to be of widespread use. The 2013 program successfully collected data from all six holes, although the data was not processed as part of this work program.

Where down-hole poor ground conditions were encountered, the oriented core tool proved to be of little value due to the incompetent nature of intensely altered and mineralized rock. Wherever competent rock was encountered, oriented core data were collected.

10.14 Down Hole Surveying

All drill holes during the 2010 drill program were surveyed using a Reflex Technologies single EZ-Shot or EZ-Trax down-hole survey system. Drill holes were surveyed once completed – this procedure was used because of the chance that bad ground conditions encountered in the drill holes increased the risk of cave-in when pulling the drill string backwards to conduct a survey. A cave-in can result in increased cost due to time spent reaming the drill hole clean back to the bottom, or from the possibility of sticking the drill string, causing loss of drilling tools. The presence of magnetite in banded iron formation and relatively unaltered trachyte or greenstone caused problems with respect to azimuth readings and also the azimuth of the drill traces. This required many repetitions of the down-hole survey readings, which in some cases resulted in an inability to record consistent data.

For the 2011, 2012, and 2013 programs, the Reflex Down-Hole Gyro survey system was adopted with the EZ-Trax or EZ-Shot down-hole survey tools as back up. The Reflex Gyro is built around a digital micro-gyro, which consists of a silicon sensor chip and an integrated circuit assembled in a ceramic (non-magnetic) package. The gyro provides directional data (azimuth and dip) at any interval from inside the drill rods. This system is used to provide azimuth and inclination data in rocks with strong magnetic fields, because the gyros operate independently of the earth's magnetic field. The system also records ambient temperature as well as collecting basic gravity measurements. The Reflex gyro system was successfully applied to the majority of the 2011 and 2013 drill programs, as well as to the 2016 and 2020 drill programs completed by First Mining.

Data recorded from the down-hole surveys was incorporated into 3D planning and modelling.

10.15 Drilling Pattern and Density

The overall drill pattern approximates a 50 m grid along the long axis of the Portage zone and approximately 45 to 65 m spacing down the dip of the mineralized zone. SRK is of the opinion that the drill spacing, and density is appropriate for this type of deposit and style of mineralization.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

The following sections outlining sample preparation, analysis and security refer only to drill programs carried out by Gold Canyon and First Mining, and not to drilling conducted by prior operators.

11.1 Core Drilling and Sampling

Detailed descriptions of the drill core were carried out under the supervision of a senior geologist, a member in good standing of the APGO (Association of Professional Geologists of Ontario) and AIPG (American Institute of Professional Geologists). The core logging was carried out on-site in a dedicated core logging facility. Drill log data from drill programs up to 2016 were recorded onto paper logs that were later scanned and digitized. Logging of the 2018 and 2020 drill core was completed using Datamine 'DH Logger' software, and data was imported directly into First Mining's central Fusion SQL drilling database.

Core was laid out 30 to 40 boxes at a time. First, the core was photographed in 15 m batches prior to logging or sampling. This was followed by a geotechnical log that recorded quantitative and qualitative engineering data including detailed recovery data and rock quality designation. Any discrepancies between marker blocks and measured core length were addressed and resolved at this stage. The core was then marked up for sampling.

For Gold Canyon's 2010 and 2011 drill programs, and the 2016 – 2020 First Mining drill programs, all the drill core intervals were sampled using sample intervals of 1 m. During the 2012 drilling program, Gold Canyon changed its standard sample length from 1 m to 2 m lengths. However, in zones of poor recovery, 1.5 m or 3 m samples were sometimes collected. Samples over the standard sample length were typically half core samples and whole core was generally only taken in intervals of poor core recovery across the sampled interval. Sampling marks were made on the core and sample tickets were stapled into the core boxes at the beginning of each sample interval.

Quality control samples were inserted into the sample stream. Inserting quality control samples involved the addition of certified blanks, certified gold standards, and field and laboratory duplicates. Field duplicates were collected by quartering the core in the sampling facility on-site. Laboratory duplicates were collected by splitting the first coarse reject and crushing and then generating a second analytical pulp. Blanks, standards, and duplicates made up on average 10% of the total sample stream. Sample tickets were marked blank, field or laboratory duplicate, or standard, and a sample tag was stapled into the core box within the sample stream.

Geological descriptions were recorded for all core recovered. Separate columns in the log allow description of the lithology, alteration style, intensity of alteration, relative degree of alteration, sulphide percentage, rock colour, vein type, and veining density. A separate column was reserved for written notes on lithology, mineralization, structure, vein orientations/relations etc. The header page listed the hole number, collar coordinates, final depth, start/end dates, and the name of the core logging geologist.

11.1.1 Core Sampling, Handling and Chain-of-Custody

Following the logging and core marking procedures described in the previous sub-section, the core was passed to the sampling facility. Core sampling was performed by experienced sampling technicians (for Gold Canyon's drill programs, technicians were from Ackewance Exploration & Services of Red Lake, Ontario), or on-site geologists, and quality control was maintained through regular verification by on-site geologists. Core was broken, as necessary, into manageable lengths. Pieces were removed from the box without disturbing the sample tags, were cut in half lengthwise with a diamond saw, and then both halves were carefully repositioned in the box. When a complete hole was processed in this manner, one half was collected for assay while the other half remained in the core box as a witness. The remaining core in the boxes was then photographed. All logs and photographs were then submitted to the senior geologist/project manager for review and were archived. Data were backed up.

The sampling technician packed one half of the split core sample intervals into transparent vinyl sample bags that were sequentially numbered to match the sample number sequences in the sample tag booklets used by the core-logging geologists. The numbered, blank portion of the triplicate sample tag was placed in the bag with the sample; the portion that was marked with the sample interval remained stapled into the bottom of the core box at the point where the sample interval begins. Sample bags were then sealed with plastic tags. Sealed sample bags were packed into rice sacks five samples at a time. All sacks were individually labeled with the name of the company, number of samples contained therein, and the number sequence of the samples therein. Sacks were assigned sequential numbers on a per shipment basis. A project geologist then checked the sample shipment and created a shipping manifest for the sample batch. A copy was given to the Project manager and a copy was sent along with the sample shipment. A copy of the sample shipment form was also sent via e-mail to the analytical laboratory.

The Project geologist prepared the sample submission form for the assay laboratory. This form identified the number of sample sacks as well as the sequence of sample numbers to be submitted. Due to the remote location, the shipment was then loaded on to a plane or helicopter and flown direct to Red Lake where representatives of the commercial analytical laboratory met the incoming flight and took the samples to the laboratory by pickup truck.

Once at the laboratory, a manager checked the rice sacks and sample numbers on the submission form. The laboratory then split the received sample manifest into batches for analysis, assigned a work order to the batch, and sent a copy of the mineral analysis acknowledgement form to the Project manager.

Aluminum tags embossed with the hole number, box number, and box interval (from/to) were prepared and stapled onto the ends of each core box. Core boxes were cross stacked on pallets and then moved to on-site storage.

11.2 Sample Security

Core samples collected at the drill site were held in closed core boxes sealed with fiber tape; at various times of day, camp staff collected the core boxes that were then delivered to the core logging facility. All core logging, sampling and storage took place at the Springpole Gold Project site. Following the logging and marking of core (described in the preceding section), all core preparation and sampling was performed by technicians (for Gold Canyon's drill programs, technicians were from Ackewance of Red Lake, Ontario) under the supervision of the Project manager, or by company geologists. All on-site sampling activities were directly supervised by the Project manager or geologist.

11.3 Sample Preparation and Analytical Procedures

11.3.1 Analytical Laboratories

All gold assay work since the 2010 drill program has been performed by SGS Laboratories in Red Lake, Ontario. Silver and multi-element assays for the Gold Canyon drill programs were performed by the SGS Don Mills laboratory in Toronto, Ontario, and by the SGS laboratory in Vancouver for First Mining's 2016 and 2020 drill programs. The SGS facilities are certified and conform to requirements CAN-P-1579 and CAN-P-4E (ISO/IEC 17025:2005). Certification is accredited for precious metals including gold and silver and 52 element geochemical analyses.

First Mining has attested that there is no commercial nor other type of relationship between First Mining and SGS Laboratories that would adversely affect the independence of SGS Laboratories.

11.3.2 Analytical Procedures

All samples received by SGS Red Lake were processed through a sample tracking system that is an integral part of the company's laboratory information management system. This system utilizes bar coding and scanning technology that provides complete chain of custody records for every stage in the sample preparation and analytical process.

Samples were dried, and then crushed to 70% of the sample passing 2 mm (-70 mesh). A 250 g sample was split off the crushed material and pulverized to 85% passing 75 microns (200 mesh). A 30 g split of the pulp was used for gold fire assay and a 2 g split was used for silver analysis. Crushing and pulverizing equipment was cleaned with barren wash material between sample preparation batches and, where necessary, between highly mineralized samples. Sample preparation stations were also equipped with dust extraction systems to reduce the risk of sample contamination. Once the gold assay was complete, a pulp was sent to the SGS Toronto facility for silver and possibly for multi-element geochemical analysis.

As part of the standard internal quality control procedures used by the laboratory, each batch of 75 Springpole core samples included four blanks, four internal standards, and eight duplicate samples. In the event that any reference material or duplicate result would fall outside the established control limits, the sample batches would be re-assayed.

Pulps and rejects from the First Mining core samples, as well as from earlier drill programs where still available, are currently being kept in storage by First Mining.

11.3.3 Gold, Silver and Multi-Element Analysis

Prepared samples were analyzed for gold by fire assay with atomic absorption finish. Samples returning assays in excess of 10 g/t Au were re-analyzed with a gravimetric finish.

Prepared pulp samples shipped from SGS Red Lake to SGS Toronto were analyzed for silver by three-acid digestion with atomic absorption finish.

During the winter 2010 program, prepared samples were analyzed for 52 elements by acid digestion (3:1 HCl : HNO_3). The list of elements is included in Table 11-1.

All samples from the 2016 and 2020 drill programs by First Mining were also analyzed for 52 elements by acid digestion (ICM14B).

Table 11-1: SGS Multi-Element Analysis Method ICM14B – Detection Limits

Element	Limits	Element	Limits	Element	Limits
Ag	0.01 – 10 ppm	Hg	0.01 ppm - 1%	Se	1 ppm - 0.1%
Al	0.01 - 15%	In	0.02 ppm - 0.05%	Sn	0.3 ppm - 0.1%
As	1 ppm - 1%	K	0.01 - 25%	Sr	0.5 ppm - 1%
B	10 ppm - 1%	La	0.1 ppm - 1%	Ta	0.05 ppm - 1%
Ba	5 ppm - 1%	Li	1 ppm - 5%	Tb	0.02 ppm – 1%
Be	0.1 ppm - 0.01%	Lu	0.01 ppm - 0.1%	Te	0.05 ppm - 0.1%
Bi	0.02 ppm - 1%	Mg	0.01 - 15%	Th	0.1 ppm - 1%
Ca	0.01 - 15%	Mn	2 ppm - 1%	Ti	0.01 - 15%
Cd	0.01 ppm - 1%	Mo	0.05 ppm - 1%	Tl	0.02 ppm - 1%
Ce	0.05 ppm - 0.1%	Na	0.01 - 15%	U	0.05 ppm - 1%
Co	0.1 ppm - 1%	Nb	0.05 ppm - 0.1%	V	1 ppm - 1%
Cr	1 ppm - 1%	Ni	0.5 ppm - 1%	W	0.1 ppm - 1%
Cs	0.05 ppm - 0.1%	P	50 ppm - 1%	Y	0.05 ppm - 1%
Cu	0.5 ppm - 1%	Pb	0.2 ppm - 1%	Yb	0.1 ppm - 0.01%
Fe	0.01% - 15%	Rb	0.2 ppm - 1%	Zn	1 ppm - 1%

11.4 Bulk Density Data

Bulk density was obtained for select core samples using the paraffin wax method. For the Gold Canyon samples, this was undertaken by SGS Lakefield Research Ltd. laboratory in Lakefield, Ontario. For the additional samples taken in 2020 by First Mining, bulk density testing was undertaken by the SGS laboratory in Red Lake, Ontario.

The bulk density of a sample is the weight of the sample divided by the volume of the sample including voids.

The procedure as applied by SGS laboratory was as follows:

1. Oven-dry the samples and then cool to room temperature
2. Label and weigh each sample in grams
3. Coat the sample with paraffin wax heated in a container immersed in boiling water
4. Repeatedly immerse the sample in the wax until completely sealed
5. Avoid heating the sample
6. Weigh the waxed sample and record

7. Weigh the waxed samples (g) by suspending in water and recording the displaced volume (mL) and the water temperature (ºC)
8. Remove the wax by placing in boiling water or freezing the core and chipping off if return of the sample is required

Calculations:

1. Weight of wax = (weight of sample + wax) – (weight of sample)
2. Volume of wax = weight of wax /specific gravity (SG) of wax corrected for temperature.
3. Volume of sample = (volume of sample + wax) – (volume of wax)
4. Bulk density (t/m^3) = weight of sample (g) / volume of sample (mL)
5. Bulk Density (lb/ft^3) = (t/m^3) / 0.0160

Results from selected analysis of bulk density are discussed in Section 14.10.2 of the Report and a summary shown in Table 11-2.

Table 11-2: Summary of Wax Bulk Density Measurements

Sample					Weight (g)				Volume (cm^3)			Rock Density	
No.	Description		Box No	m	Dry Rock	Rock Coated with wax	Weight in Water	Water Displacement	Rock Coated	Volume Wax	Rock	Density (g/cm^3)	Density (lbs./ft^3)
1	SP11-061	1	1	40.5 - 59	804.0	815.2	497.2	513.0	318	12.5	305.8	2.63	164.2
2		2	8	81.7 - 85.5	523.1	533.1	223.1	310.0	310	11.2	299.1	1.75	109.2
3		3	13	98.2 - 101	219.7	227.5	95.1	132.4	133	8.8	123.8	1.77	110.2
4		4	18	113.9 - 114.2	397.1	407.3	192.8	214.5	215	11.4	203.3	1.95	122.0
5		5	24	131.2 - 132	517.7	528.7	262.5	255.2	265	12.3	253.1	2.05	127.7
6	SP11-065	1	21	590.1 - 52.6	805.0	820.3	522.9	297.4	298	17.2	280.5	2.87	179.2
7		2	37	104.2 - 105.8	1042.4	1052.1	665.1	397.0	397	22.1	375.3	2.78	173.4
8		3	49	138.4 - 140.5	662.3	684.0	408.0	276.0	276	24.3	251.9	2.63	164.1
9		4	72	199.3 - 201.8	634.5	650.6	363.6	287.1	287	18.0	269.5	2.36	147.0
10		5	82	226.4 - 229.6	769.8	793.0	497.1	325.9	325	20.9	300.2	2.56	160.1
11	SP11-069	1	97	265.5 - 267.2	871.5	895.5	512.3	383.2	304	26.9	356.7	2.44	152.6
12		2	108	295.1 - 297.2	532.7	546.7	271.3	275.4	275	15.7	260.0	2.05	127.9
13		3	130	345.2 - 346.7	630.7	647.5	354.3	283.2	283	15.8	264.6	2.38	148.8
14		4	146	376.8 - 379.1	625.4	641.5	368.3	273.2	273	16.1	255.4	2.45	152.9
15		5	158	401.9 - 403.9	826.4	847.1	520.6	326.5	327	23.2	303.6	2.72	169.9

11.5 Quality Assurance and Quality Control Programs

11.5.1 Pre-2007 QA/QC Program

No documentation relating to sample handling and preparation or sample QA/QC documentation for the pre-2003 drilling was provided to the author.

A total of 1,725 database entries were checked against the original certificates. Only a few data entry errors were observed and corrected; however, the total number of errors was not reported.

The QA/QC program for 2003 to 2007 consisted of:

- resubmission of approximately 10% of the sample pulps to a second laboratory (ALS Chemex)
- insertion of two commercial standard reference materials (standards submitted every 30th sample)
- insertion of blanks

There were no field or coarse reject duplicates submitted. Also, no pulp duplicates were submitted to the primary laboratory.

Due to the lack of detailed documentation, particularly for pre-2003 drilling, in 2013, Gold Canyon decided to initiate a core resampling program focusing on the pre-2003 drilling. Specifically, Gold Canyon carried out silver assays for sections that were previously not assayed for silver, expanded the assay intervals to include previously unsampled intervals in the older core and carried out an extensive re-sampling of the previously sampled core to improve on the quality control procedures. A total of 3,352 samples were collected for assays, these include 2,768 check assays for gold, 2,179 check assays for silver, 457 new gold assays and 1,173 new silver assays. SRK reviewed the results of the re-sampling program and concluded that the results did not identify any bias with the pre-2003 drilling and that the pre-2003 drilling was acceptable for inclusion in the resource estimate. The East Extension and Camp zones as now defined correspond to the deposits estimated by P&E in their 2006 study (Armstrong, 2006).

11.5.2 2007/2008 QA/QC Program

A total of 18 drill holes were completed in 2007 and 2008 comprising a total of 1,374 assay intervals. These samples were assayed for gold only by the Accurassay Laboratories of Thunder Bay, Ontario. SRK checked a total of 137 samples representing 10% of the total against the original certificates. No errors were found.

No program was set up for duplicates, standards, or blanks for this drilling program. The laboratory ran their own set of duplicates for internal monitoring purposes; however, those data were not available to SRK. Accurassay Laboratories was an independent laboratory. SRK is unaware if the laboratory had a specific accreditation in 2007.

11.5.3 2010 to 2013 QA/QC Program

In 2010, Gold Canyon instituted a QA/QC program consisting of commercial standard reference materials for gold, and, consistent with current industry practice, blanks, field duplicates, and pulp duplicates. In addition, a "round robin" program was instituted in 2011 with ACT Labs of Red Lake, Ontario, that compared pulp re-assay results against the original SGS results for 469 samples. ACT Labs was an independent laboratory. SRK is unaware if the laboratory had a specific accreditation in 2007.

SGS conducted their own program of internal duplicate analysis as well. The results of this program were also analyzed by SRK as a valuable comparison against the "blind" pulp duplicates submitted (SRK, 2019).

A summary of the blanks and standards submissions from the 2010 to 2013 assay programs are presented below:

- a total of 1,605 field duplicates were submitted for gold
- a total of 1,532 field duplicates were submitted for silver
- a total of 1,591 pulp duplicates were submitted for gold
- a total of 1,515 pulp duplicates were submitted for silver
- a total of 1,173 commercial gold standards were submitted from a set of 19 different commercial standards
- no commercial standards were submitted for silver
- a total of 1,647 blanks were submitted with the gold assays
- a total of 660 blanks were submitted with the silver assays

The total submissions for gold duplicates, standards and blanks was 5,387; representing 10.1% of the samples assayed for gold. The total submissions for silver duplicates and blanks was 3,667; representing 7% of the total samples assayed for silver.

11.5.4 2016 and 2020 QA/QC Programs

For the First Mining QA/QC programs from the 2016 and 2020 drilling, blanks and standards were inserted as a general rule at a rate of one standard for every 20 samples (5% of total), and one blank for every 30 samples (3% of total). 'Coarse' duplicates taken from coarse reject, and 'pulp' duplicates taken from 250 g pulverized splits, were also inserted at regular intervals with an insertion rate of 4%. For the 2020 assay program, field duplicates from quartered core were also inserted at regular intervals, with an insertion rate of 4%.

In addition to the QA/QC program implemented by First Mining, the laboratories operate their own internal laboratory QA/QC system, inserting quality control materials, blanks, laboratory replicates and laboratory duplicates on each analytical run.

Blanks made of barren garden rock purchased from a local hardware store were used. A threshold of ten times the lower detection limit (LDL) was used as a guide to determine potential contamination. Any assays above this threshold were reviewed on a case-by-case basis to determine if any corrective action was required at that laboratory. If a single blank or standard within a zone of mineralization was deemed to have failed, that QA/QC sample plus five samples either side in the same batch were sent for reanalysis. If a blank/standard plus one or more consecutive standards were deemed to have failed, then the failed samples plus ten samples either side and all the samples between, were sent for reanalysis. For samples from unmineralized zones, if a single standard failed within a batch where the other standards or blanks passed, the entire batch was deemed to have passed and no corrective action was taken.

Ten different gold standards were used in the 2016 and 2020 programs, spanning a range of gold grades from 0.06 ppm to 9.59 ppm, as summarized in Table 11-3. Two of these standards were also used as silver standards in the 2020 assay program. The standards were supplied by CDN Resource Laboratories Ltd. (CDN) of Vancouver, BC. A standard was deemed suspect as a failure if the result fell outside 3 standard deviations from its expected value as defined by the standard's certificate. Any assays outside of this threshold were reviewed on a case-by-case basis to determine if any corrective action was required.

Table 11-3: List of Standards used in 2016 and 2020 Assay Programs

Drill Program	Standard ID	Method	Au (ppm)	2 St. Dev (ppm)	Ag (ppm)	2 St. Dev (ppm)	Lab Name
2016	CDN-GS-1R	30g Fire Assay	1.21	0.11	-	-	CDN
2016	CDN-GS-2M	30g Fire Assay	2.21	0.244	-	-	CDN
2016	CDN-GS-3P	30g Fire Assay	3.06	0.18	-	-	CDN
2016	CDN-GS-5R	30g Fire Assay	5.29	0.34	-	-	CDN
2016	CDN-GS-10E	30g Fire Assay	9.59	0.53	-	-	CDN
2016	CDN-GS-P4F	30g Fire Assay	0.498	0.056	-	-	CDN
2020	CDN-GS-1W	30g Fire Assay	1.063	0.076	-	-	CDN
2020	CDN-GS-5G	Au: 30g Fire Assay; Ag: 4-acid digestion	4.77	0.40	101.8	7.0	CDN
2020	CDN-GS-6B	30g Fire Assay	6.45	0.33	-	-	CDN
2020	CDN-GS-P6C	Au: 30g Fire Assay; Ag: 4-acid digestion	0.767	0.078	66	5.5	CDN

A summary of the blanks, standards, and duplicate submissions from the 2016 and 2020 assay programs are presented below:

<u>2016 QA/QC Program:</u>

The total submissions for gold duplicates, standards and blanks in the 2016 assay program was 246, representing 15.7% of the samples assayed for gold. Coarse and pulp duplicates were taken at the rate of one in approximately every 25 samples, alternating between coarse and pulp within the sample stream.

- A total of 162 duplicates were submitted for gold, comprising 81 coarse duplicates and 81 pulp duplicates.
- No duplicates were submitted for silver.
- No field duplicates were taken in the 2016 program.
- A total of 41 commercial gold standards were submitted from a set of 6 different commercial standards.
- A total of 43 blanks were submitted with the gold assays.
- No commercial standards or blanks were submitted for silver.

Overall, the laboratory performed well during the 2016 assay program.

2020 QA/QC Program:

The total submissions for gold duplicates, standards and blanks in the 2020 assay program were 219 samples, representing 19.5% of the samples assayed for gold. The total submissions for silver duplicates and blanks was 192 samples, representing 17.1% of the total samples assayed for silver.

Field duplicates were taken approximately every 30 samples, and coarse and pulp duplicates were taken at the rate of one in approximately every 25 samples, alternating between coarse and pulp within the sample stream.

- A total of 58 coarse and pulp duplicates were submitted for both gold and silver (30 coarse duplicates and 28 pulp duplicates).
- A total of 56 field duplicates were submitted for both gold and silver.
- A total of 60 commercial gold standards were submitted from a set of 4 different commercial standards.
- A total of 33 commercial silver standards were submitted from a set of 2 different commercial standards.
- A total of 45 blanks were submitted with both the gold and silver assays.

Overall, the laboratory performed well during the 2020 assay program.

11.6 SRK Comments

In the opinion of SRK, the sampling preparation, security and analytical procedures used in the drill programs conducted by Gold Canyon for gold analyses are acceptable but not fully consistent with generally accepted industry best practices because of the lack of standard reference material for silver for the earlier drill campaigns. However, because of the relatively low economic value of silver, SRK concludes that the assay data are adequate for use in resource estimation. First Mining has an established QA/QC protocol for the acceptance of assay batches with respect to the performance of standard reference material, duplicates, and blanks. SRK also notes that First Mining is now including some standard reference material for silver in their drill programs.

12 DATA VERIFICATION

Of the 18 drill holes completed in 2007 and 2008, comprising a total of 1,374 assay intervals analyzed for gold, SRK checked a total of 137 samples representing 10% of the total against the original certificates. No errors were found.

A total of 3,135 assay values for gold and 3,161 assay values for silver in the database were compared against the original protected PDF assay certificates submitted by SGS Red Lake. These totals represent 10.1% and 10.4% of the total number of assays for gold and silver, respectively.

Of the original assay values checked against certificates, the focus was on values material to any resource estimate, either higher-grade intervals or very low-grade intervals in proximity to higher-grade intervals. The average grade of gold samples verified was 2.05 g/t Au. The average grade of silver samples checked was 8.27 g/t silver.

Only two errors were found for gold:

- The gold value of sample interval SP10-028 from 433 m to 436 m (sample number 8287) was found to have an entered value of 5.96 g/t Au against a value on the assay certificate of 9.00 g/t Au.
- The gold value of sample interval SP11-076 from 69 to 70 m (sample number 14583) having the value of 0.45 oz/t was incorrectly placed in the parts per billion column.

No errors were found with respect to silver assays.

This represents an error rate of 0.064% in gold assays and an error rate of 0.0% in silver assays. This error rate is well within acceptable industry standards.

12.1 Verifications by SRK

12.1.1 Site Visit

SRK carried out visits to the Springpole site on February 10 and 11, 2012, and again on August 8 and 9, 2012. During the site visits, core logging procedures were reviewed. Several sections of core from the Portage, Camp, and East Extension zones were examined. Sampling procedures and handling were observed. The deposit geology, alteration, and core recovery data were observed for the Portage zone. SRK was fully assisted during the site visits by Springpole personnel and was given full access to data during the site visits. Springpole field personnel were very helpful and fully cooperative during both site visits.

During the site visits, SRK re-logged mineralized sections of drill core from the Springpole deposit and checked geological units against the recorded written logs. Down-hole survey data entered in the digital database was checked against data entered on paper logs at the site and no errors were noted. Drill site locations could not be verified as most drill sites are situated under Springpole Lake, but SRK did observe two drill platforms drilling on Springpole Lake during the visit.

12.1.2 Verifications of Analytical Quality Control Data

As part of the mineral resource estimation process, SRK reviewed the QA/QC data collected by Gold Canyon, reviewed the procedures in place to assure assay data quality, and verified the assay database against original assay certificates provided directly to SRK by SGS Red Lake, the assay laboratory. A total of 53,431 gold assays, 46% of the assay data, were checked against original assay certificates. No significant database errors were identified. About 143 minor rounding errors were observed. None of the rounding errors are deemed material or of any significance to the mineral resource estimate presented in this report.

12.1.3 Independent Verification Sampling

A total of three mineralized quarter core samples were collected during the February 2012 site visit. The intent of the sampling program was only to determine if gold did occur in concentrations similar to what had been reported by Gold Canyon. Assays from the samples collected by SRK are presented in Table 12-1. The re-sampling agrees with the original Gold Canyon sampling.

Table 12-1: Assays from Duplicated Samples Collected During Site Visit

SRK Check Assay		Gold Canyon Original	
Sample	Au (g/t)	Sample	Au (g/t)
9135	8.64	9135	9.04
9136	7.49	9136	7.85
6152	2.37	6152	2.77

12.2 Authors' Comments

In the opinion of the QP, the integrity of the sample data for the Springpole Gold Project is adequate for inclusion in mineral resource estimation and for the purpose that it is used in this Report.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

The Springpole deposit has been the subject of several metallurgical testwork programs and previous studies, as summarized in Section 13.1. Two different flowsheets were compared: i) whole ore leaching and ii) rougher/cleaner sulphide flotation followed by separate leaching of the concentrate and tailings streams.

Earlier flotation testwork campaigns (SRK, 2013; SRK, 2019) resulted in a high mass pull to concentrate, requiring high power demand for concentrate regrinding. This was attributed to high feldspar and mica content and non-sulphide gangue reporting to the flotation concentrate. During the 2019 Preliminary Economic Assessment (PEA) for the Project, investigations were made into rejecting mica using flotation and/or hydraulic classification to reduce the comminution power requirement and further concentrate the ore. However, poor recoveries resulted from these modifications and conventional flotation was favoured in the recommended flowsheet of flotation followed by separate leaching of both concentrate and tailings.

During the 2020 PFS, the testwork program focused on conventional flotation and leach process flowsheet comprising:

- primary grind 80% passing (P_{80}) size of 150 μm
- conventional flotation – roughing and cleaning
- concentrate regrind and leaching
- flotation tails leaching
- cyanide detoxification
- solid-liquid separation

This was compared with whole ore leaching flowsheet results to further cross-check capital cost saving opportunities.

13.1 Historical Testwork Programs

A summary of historical testwork campaigns is presented in Table 13-1. Further detail can be found in the 2019 Updated PEA technical report for the Project (SRK, 2019).

Table 13-1: Summary of Historical Testwork

Year	Laboratory	Testwork Performed
1989	Lakefield Research, Lakefield; LR3657	Whole ore leach cyanide leach and CIL
2011	SGS Mineral Services, Vancouver; 50138-001	Whole ore cyanide leach
2013	SGS Mineral Services, Lakefield; 13152-001	Whole ore cyanide leach Flotation and concentrate regrind leach
2013	Process Mineralogical Consulting Ltd; Oct2013-05	Mineralogical analysis of two grab samples
2017	Base Met Labs, Kamloops; BL0161	Comminution testing Mineralogical assessment – BMA, TMS Whole ore leach Rougher flotation and concentrate regrind leach Viscosity
2018	ALS Metallurgy, Kamloops; 180107	Whole ore cyanide leach Flotation: Concentrate regrind leach and tail leach
2018	Jacobs Engineering Group, Lakeland Florida	Reverse flotation to float off mid-size mica to reduce comminution requirement
2018	Eriez Flotation Division, Erie Pennsylvania	Hydraulic classification to remove multiple size fractions of micas to reduce comminution requirement – cross flow and hydrofloat separation

13.2 2020 SGS Testwork Program

The 2020 SGS testwork program was split into two phases:

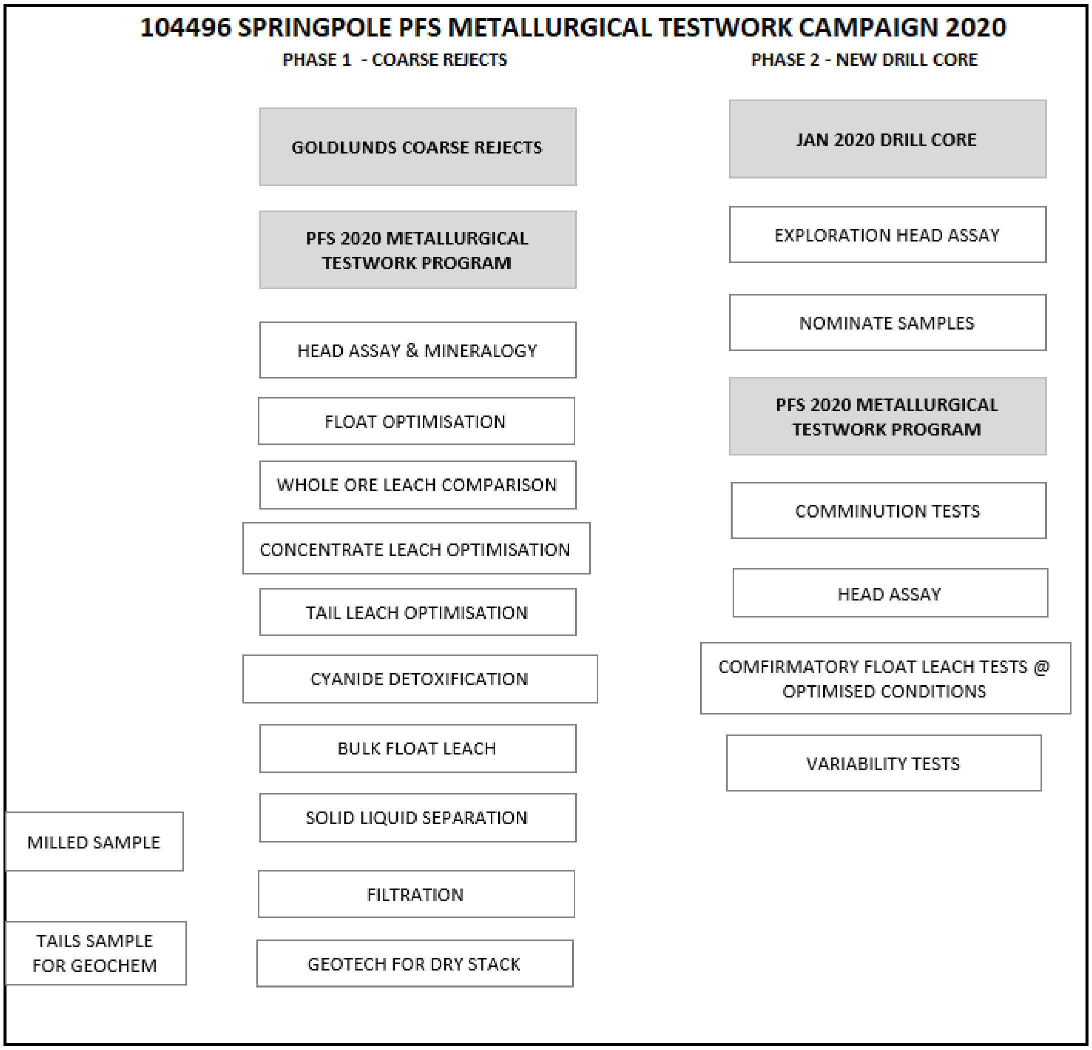
- Phase 1 using coarse reject samples from the 2016 drilling campaign
- Phase 2 using new HQ drill core from the 2020 drilling campaign

The PFS testwork scope is summarized in Figure 13-1. Both phases involved complete head analysis of samples while Phase 1 included quantitative mineralogy: bulk composition and gold deportment/association.

Phase 1 focused on comparing whole ore leach testwork results with optimized conditions for flotation + concentrate/tailings leaching tests. A bulk float + leach test was done to generate large concentrate and tailings samples masses for downstream testing (e.g. regrind power requirements and solid/liquid separation). Geotechnical characterization testwork was also included in Phase 1 to support the design of the dry stack tailings facility.

Phase 2 included additional comminution tests on the larger drill core size as well as confirmatory float + leach tests under the optimized conditions. Twelve variability samples were also to be evaluated on the final, recommended PFS flowsheet.

Figure 13-1: Springpole PFS Metallurgical Testwork Campaign 2020 – Phase 1 and Phase 2 Programs



Source: SRK, 2021

13.2.1 Sample Selection

Phase 1 coarse reject samples were delivered to SGS Lakefield in January 2020. Four zone composites were selected based on spatial zone, head grade, and lithology for the Phase 1 metallurgical testwork campaign as presented in Table 13-2.

Table 13-2: Phase I Sample Composition

Composite	Nominal Depth	Drill hole	Lithology	Intervals (m)
Composite 1	Mid-upper	PM-DH-01	Tuff, porphyry	22-25, 27-34.5, 36-49, 88-98, 99-108
		PM-DH-02		73-75, 90-96, 97-100,163-171
Composite 2	Mid	PM-DH-02	Porphyry, breccia, tuff	189-204, 221-226, 228-233, 243-253
		PM-DH-03		209-217, 321-325
Composite 3	Mid-upper	PM-DH-03	Trachyte	75-86, 87-90, 194-209
Composite 4	Lower	PM-DH-03	Trachyte	254-268, 283-294

Phase 2 drill core was delivered to SGS Lakefield in May and August 2020. Four more zone composites were selected based on spatial zone, head grade, and lithology as presented in Table 13-3.

Table 13-3: Phase 2 Sample Composition

Composite	Nominal Depth	Drill hole	Lithology	Intervals (m)
Composite 5	Upper	SM20-001	Trachyte	11-19, 23-28
Composite 6	Lower	SM20-002	Trachyte	237-254
Composite 7	Lower	SM20-003	Breccia	284-298
Composite 8	Mid	SM20-003	Porphyry	130-141

Variability samples were selected based on discrete intervals according to lithology and head grade as presented in Table 13-4. Var 2 and Var 5 had insufficient mass in the selected intervals and were dropped from the list of tested samples.

Table 13-4: Phase 2 Variability Samples

Composite	Variability	Drill hole	Lithology	Intervals (m)
Variability 1	Low grade	SM20-001	Trachyte	31-37
Variability 2	Low grade	SM20-001	Breccia	137-144 (insufficient)
Variability 3	Low grade	SM20-002	Trachyte	129-136
Variability 4	Mid-grade	SM20-003	Porphyry	192-199
Variability 5	High grade	SM20-003	Porphyry	216-222, 225-227 (insufficient)
Variability 6	High grade	SM20-003	Breccia	259-267
Variability 7	Low grade	SM20-001	Trachyte	116-121
Variability 8	High grade	SM20-001	Trachyte	294-300
Variability 9	Mid-grade	SM20-002	Breccia	85-89
Variability 10	Mid-grade	PM DH-04	Andesite	7–27
Variability 11	Mid-grade	PM DH-04	M Sed	204-221
Variability 12	Mid-grade	PM DH-04	Breccia	305-321

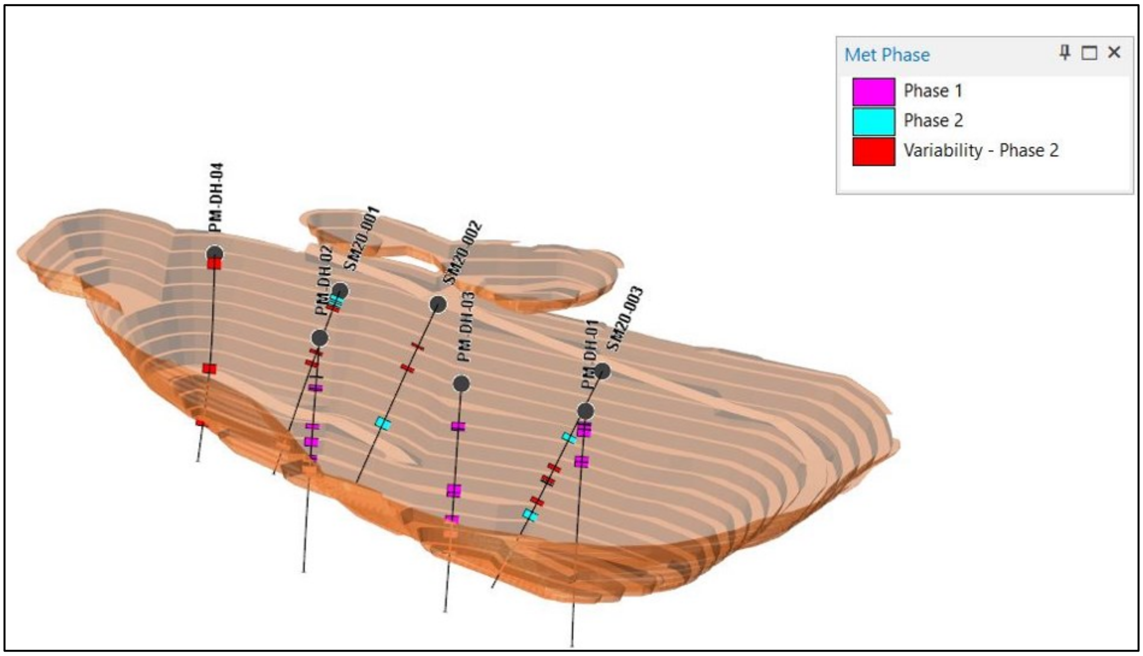
Twenty-four composite samples were prepared for comminution testwork as shown in Table 13-5. In addition, three of the metallurgical variability samples (Var 10 to 12) were also tested for ore competency and hardness/grindability characteristics.

Table 13-5: Comminution Samples – Phase 2

Composite	Description	Drill hole	Lithology	Intervals (m)
Tra 1-Ore A	Drill Core Samples	SM20-001	Trachyte	11-28
Tra 1-Ore B	Drill Core Samples	SM20-001	Trachyte	227-242
Tra 1-Ore C	Drill Core Samples	SM20-001	Trachyte	293-310
Tra 1-Ore D	Coarse Rejects	SM20-001	Trachyte	121-134
Tra 2-Ore A	Drill Core Samples	SM20-002	Trachyte	248-263
Tra 2-Waste A	Coarse Rejects	SM20-002	Trachyte	11-28
Tra 3-Ore A	Drill Core Samples	SM20-003	Trachyte	142-143
Tra 3-Ore B	Drill Core Samples	SM20-002	Trachyte	284-298
Tra 3-Ore C	Coarse Rejects	SM20-003	Trachyte	380-396
Tra 3-Ore D	Coarse Rejects	SM20-003	Trachyte	340-357
Tra 3-Ore E	Coarse Rejects	SM20-003	Trachyte	407-430
BX 1-Ore	Drill Core Samples	SM20-001	Breccia	134-143
BX 1-Waste	Coarse Rejects	SM20-001	Breccia	155-165
BX 2-Ore A	Drill Core Samples	SM20-002	Breccia	283-295
BX 2-Ore B	Drill Core Samples	SM20-002	Breccia	339-354
BX 2-Waste A	Coarse Rejects	SM20-002	Breccia	78-93
BX 2-Waste B	Coarse Rejects	SM20-002	Breccia	151-166
BX 2-Ore C	Coarse Rejects	SM20-002	Breccia	365-380
BX 3-Ore A	Drill Core Samples	SM20-003	Breccia	259-276
Msed 3-Waste	Coarse Rejects	SM20-003	Metasedimentary	14-15
Por 3-Ore A	Drill Core Samples	SM20-003	Porphyry	197-212
Por 3-Ore B	Drill Core Samples	SM20-003	Porphyry	216-235
Por 3-Ore C	Drill Core Samples	SM20-003	Porphyry	315-330
Por 3-Waste	Coarse Rejects	SM20-003	Porphyry	74-84

The location of the selected drillhole intervals for the two testwork phases and composite versus variability samples is shown in Figure 13-2 relative to the final pit shell outline.

Figure 13-2: PFS Metallurgical Testwork Sample Selection within Pit Shell



Source: SRK, 2021

13.2.2 Sample Head Analysis

Phase 1 and 2 composites were submitted for a full suite of assays including:

- Au, Cu, As, Hg by direct assay
- Sulphur (S_{total}, sulphide sulphur S^{2-})
- Carbon (C_{org}, C_{total})
- ICP scan for 18 elements

Key assays for the composites tested are shown in Table 13-6.

Table 13-6: Summary of Phase 1 and Phase 2 Composite Head Assays

Element	Unit	Comp 1	Comp 2	Comp 3	Comp 4	Comp 5	Comp 6	Comp 7	Comp 8
Au	g/t	1.07	1.33	1.26	1.08	1.30	0.71	0.80	1.01
Ag	g/t	6.0	6.2	7.3	6.8	3.5	4.6	5.1	18.0
Cu	%	0.02	0.01	0.01	0.02	0.01	0.01	0.01	0.03
Fe	%	4.5	7.4	8.8	7.2	6.0	2.8	3.0	4.9
Zn	g/t	192	172	192	231	158	92	169	7080
Pb	g/t	83	146	81	107	58	122	156	3260
Ni	g/t	28	51	58	60	66	<20	<20	35
As	g/t	<30	<30	<30	<30	<30	<30	<30	<30
Sb	g/t	<20	<20	<20	<20	<20	<20	<20	<20
Hg	g/t	0.5	0.4	0.5	0.5	0.4	0.5	0.3	8.1
C_{total}	%	0.16	0.29	0.13	0.10	1.14	0.06	0.14	0.29
C_{org}	%	0.05	0.09	0.12	0.09	0.16	<0.05	<0.05	<0.05
S_t	%	2.72	3.13	5.63	3.98	3.72	2.62	0.17	4.22
S^{2-}	%	2.53	3.00	5.50	3.91	3.29	2.31	0.14	4.22
Te	g/t	9	10	17	11	<4	7	8	18

Observations from the head assay results for both Phase 1 and 2 composites:

- gold assays ranged from 0.71 to 1.33 g/t
- silver assays ranged from 3.5 to 18 g/t
- all samples assayed low levels of Cu, and Ni, with low to mid-levels of Pb, which contribute to cyanide consumption. Composite 8 was particularly high in Zn and Pb
- most sulphur occurs as sulphide sulphur. Sulphide sulphur assays ranged from 0.14 to 5.5%
- all samples had low levels of organic carbon indicating low potential of preg-robbing
- composite sample 8 showed high mercury content 8 g/t; the remaining samples contained around 0.5 g/t. These levels warrant mercury control
- tellurium assays ranged <4 to 18 g/t. This is consistent with the mineralogy of Springpole telluride ore

Phase 2 Variability sample head assays are presented in Table 13-7.

Table 13-7: Summary of Phase 2 Variability Sample Head Assays

Element	Unit	Var 1	Var 3	Var 4	Var 6	Var 7	Var 8	Var 9	Var 10	Var 11	Var 12
Au	g/t	0.60	0.69	0.59	1.39	0.59	2.01	1.45	1.07	1.06	0.69
Ag	g/t	2.9	2.6	2.5	12.4	3.6	16	19.5	<0.5	2.2	1.5
Cu	%	0.00	0.00	0.01	0.01	0.00	0.16	0.01	0.00	0.12	0.00
Fe	%	3.2	3.3	5.2	4.7	2.4	3.4	1.3	9.5	9.2	7.0
Zn	g/t	224	1750	151	768	73	671	2390	70	129	103
Pb	g/t	95	832	70	314	119	290	608	<40	<40	<40
Ni	g/t	43	<20	31	49	21	25	<20	23	93	25
As	g/t	<30	<30	<30	<30	<30	<30	<30	<30	<30	<30
Sb	g/t	<20	<20	<20	<20	<20	<20	<20	<20	<20	<20
Hg	g/t	0.4	3.6	<0.3	1.2	<0.3	2.7	5.7	<0.3	<0.3	<0.3
C_{total}	%	0.02	1.20	0.52	0.16	0.03	0.02	1.14	2.21	1.65	0.44
C_{org}	%	<0.05	<0.05	<0.05	0.07	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
S_t	%	2.20	2.51	3.98	3.54	1.68	3.07	0.54	1.40	5.80	4.85
S^{2-}	%	2.05	2.09	3.32	3.12	1.42	2.69	0.41	1.36	4.93	4.55
Te	g/t	<4	5	6	13	<4	15	14	4	6	<4

Observations from the Phase 2 Variability sample assays include:

- gold assays ranged from 0.60 to 2.0 g/t
- silver assays ranged from <0.5 to 20 g/t
- all samples assayed low levels of Cu, and Ni, with low to mid-levels of Pb, which contribute to cyanide consumption. Var 9 was particularly high in Zn
- most sulphur occurs as sulphide sulphur. Sulphide sulphur assays ranged from 0.41 to 4.9%
- all samples had very low levels of organic carbon, with the majority below detection limit, indicating low potential of preg-robbing
- Var samples 3, 6, 8 and 9 showed high mercury content ranging 1.2 to 5.7 g/t; the remaining samples contained around 0.4 g/t. These levels warrant mercury control
- tellurium assays ranged <4 to 15 g/t. This is consistent with the mineralogy of Springpole telluride ore

13.2.3 Sample Mineralogy

A full gold deportment study was completed on two composites with bulk mineralogy of Composites 2 and 4 determined using QEMSCAN, as summarized in Table 13-8. The results showed 23 to 33% of the samples were potassium feldspar with 40 to 50% present as mica minerals (biotite, muscovite, illite). Both composites are characterized by pyrite/marcasite sulphide mineralization with trace amounts of clay (kaolinite) minerals detected.

Table 13-8: Mineral Proportions for Phase 1 Composite Samples

Mineral Mass %	Comp 2	Comp 4
Pyrite / marcasite	5.25	7.76
Other sulphides	0.02	0.03
Quartz	5.53	9.09
K-Feldspar	32.7	23.0
Plagioclase	10.5	6.45
Muscovite / Illite	19.5	19.8
Biotite	20.1	29.9
Chlorite	0.53	0.66
Kaolinite	0.49	0.77
Other silicates	0.10	0.06
Fe-oxides	1.20	0.49
Rutile	1.12	1.17
Calcite	0.09	0.06
Ankerite	0.73	0.10
Siderite	1.38	0.02
Apatite	0.55	0.61
Other	0.23	0.06
Total	**100**	**100**

A gold deportment study was done on Comp 2 and Comp 4 as well (see Table 13-9) indicating:

- 5 to 12% of the gold is sub-microscopic (<0.6 micron) or refractory
- 8 to 14% of the gold is locked in the 0.6 to 11-micron size range
- 42 to 64% of the gold is exposed
- 22 to 32% of the gold is liberated

Table 13-9: Summary of Gold Deportment

Sample ID	Association	No. of Observed Grains	Average Size (µm)	Size Range (µm)	Au Grade (g/t)	Overall Au Distribution (%)
Comp 2	Liberated	84	3.2	0.6 – 12.8	0.29	21.8
	Exposed	428	2.8	0.6 – 25.1	0.86	64.3
	Locked	127	2.1	0.6 – 11.5	0.11	8.48
	Sub-gold	-	-	-	0.07	5.37
	SUM	**639**	**2.7**	**0.5 – 7.4**	**1.33**	**100**
Comp 4	Liberated	53	3.9	0.6 – 38.3	0.35	32.0
	Exposed	508	2.4	0.6 – 25.1	0.45	42.1
	Locked	153	2.2	0.6 – 6.7	0.15	14.1
	Sub-gold	-	-	-	0.13	11.7
	SUM	**714**	**2.4**	**0.5 – 7.4**	**1.08**	**100**

The gold distribution by gold carrier minerals by grain size is presented in Table 13-10. This shows a host of telluride minerals exist in the microscopic size range, with petzite the most dominant. Gold and electrum occur in minor amounts (<10%). Pyrite is the dominant carrier in the sub-microscopic size range.

Table 13-10: Summary of Gold Carrier Minerals

Comp 2	Gold Carrier Minerals	Gold Distribution (%)	Comp 4	Gold Carrier Minerals	Gold Distribution (%)
Mic-Au	Petzite	49.6	Mic-Au	Petzite	45.2
	Au-Ag-Pb-Te	14.4		Au-Ag-Pb-Te	16.6
	Pb-Petzite	7.69		Sylvanite	11.3
	Sylvanite	6.27		Electrum	5.58
	Electrum	5.58		Pb-Petzite	2.01
	Au-Hessite	4.01		Muthmannite	1.73
	Au	3.72		Au	1.47
	Au-Ag-Te	1.04		Au-Hessite	1.00
	Au-Pb-Hessite	0.89		Pb-Sylvanite	0.75
	Muthmannite	0.60		Calaverite	0.47
	Calaverite	0.37		Au-Ag-Te	0.45
	Au-Altaite	0.28		Au-Ag-Hu	1.74
	Au-Ag-Hg	0.17		-	-
Sub-Au	Pyrite	5.04	Sub-Au	Pyrite	11.4
	Chalcopyrite	0.01		Chalcopyrite	0.05
	Hematite	0.32		Hematite	0.23
Total		**100**			**100**

13.2.4 Comminution

Two comminution test programs were performed, including characterization tests and flotation concentrate regrind tests. To map material competency and hardness/grindability, the following comminution testwork was carried out on the samples listed in Table 13-5.

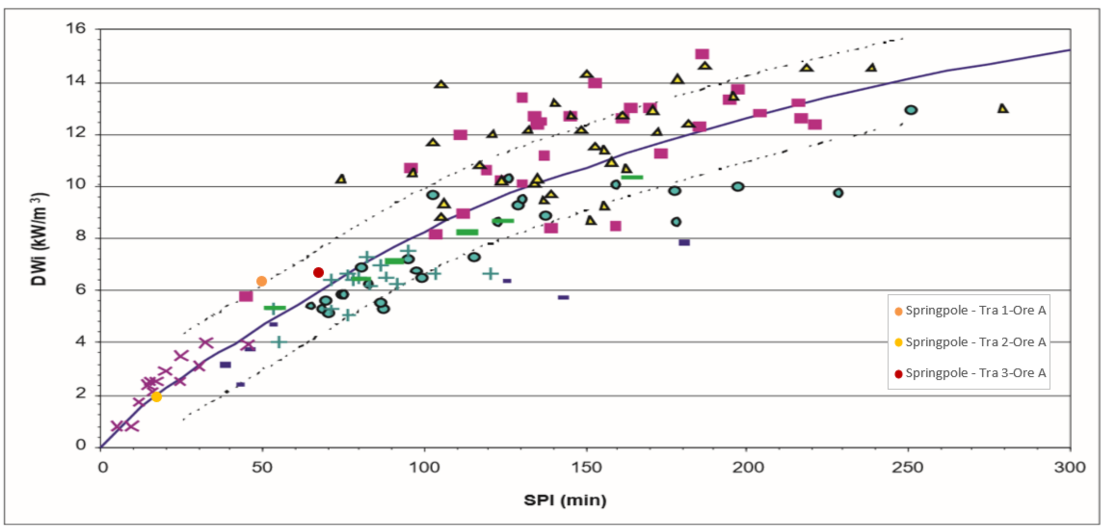
Due to the crumbly nature of some of the drill core samples, it was not possible to obtain sufficient representative particles to conduct SAG mill comminution (SMC) tests for most of the Springpole samples. Hence, the SAG power index (SPI®) test was conducted, as this test uses whole samples rather than selected size fractions.

Three parallel SMC and SPI tests were conducted to confirm a relationship between the results of these two tests. As shown in Figure 13-3, the parallel test results indicate Springpole samples align with the relationship developed from the JKTech database. The samples are categorized as 'very soft' to 'medium' competency (hardness with respect to impact resistance). Table 13-11 summarizes the results of the SMC tests.

The SPI test was performed on 13 samples, with crusher index (Ci) measurements obtained during the SPI feed preparation procedure (followed SGS Comminution Economic Evaluation Tool (CEET) protocols). Test results in Table 13-12 categorize the tested samples as 'very soft' to 'medium' hardness.

Conventional Bond work index tests were carried out to determine Bond rod mill work index (RWi), Bond ball work index (BWi_{150} at 150 µm closing screen size), and Bond abrasion index (Ai) values. Table 13-13 summarises the results of the Bond Work Index tests. (For some of the samples, only BWi tests were performed.) The average RWi and BWi_{150} are 11.0 kWh/t and 13.4 kWh/t, respectively; however, significant variation is observed in both indices. The average Ai is 0.117 g which also exhibits a wide range of variation (0.025 to 0.386).

Figure 13-3: SPI and DWi Relationship (adapted from Bailey et al..., 2009)



Source: SRK, 2021

Table 13-11:: Phase 2 SMC Test Results

Sample	Relative Density	Axb	Hardness Percentile	ta*	SCSE (kWh/t)	DWi (kWh/m3)	Mia (kWh/t)	Mih (kWh/t)	Mic (kWh/t)
Tra 1-Ore A	2.70	42.3	56	0.41	9.6	6.4	18.7	13.7	7.1
Tra 2-Ore A	2.46	124.0	8	1.31	6.5	2.0	8.1	4.7	2.4
Tra 3-Ore A	2.73	40.6	60	0.39	9.9	6.7	19.2	14.2	7.3

Table 13-12: SAG Power Index Test Results

Sample	CEET (Ci)	SPI® (Min)	Hardness Percentile
Tra 1-Ore A	10.9	49.6	27
Tra 1-Ore B	26.2	19.8	5
Tra 1-Ore C	29.8	14.2	3
X 1-Ore	28.8	7.2	1
BX 2-Ore A	15.1	17.8	4
BX 2-Ore B	31.8	16.3	4
Tra 2-Ore A	21.3	17.1	4
Tra 3-Ore A	4.2	67.3	44
Por 3-Ore A	19.6	37.6	17
Por 3-Ore B	30.5	31.7	12
BX 3-Ore A	29.2	22.0	6
Tra 3-Ore B	16.2	31.3	12
Tra 3-Ore C	22.3	35.7	15

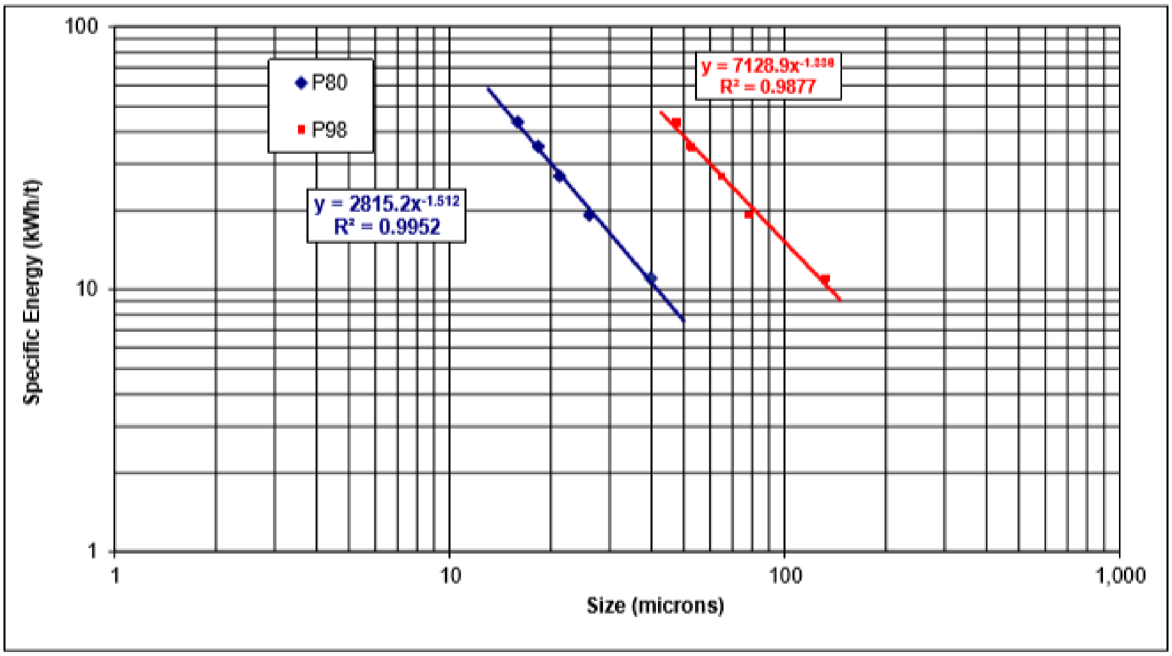
Table 13-13: Summary of Bond Test Results

Sample	RWi (kWh/t)	BWi (kWh/t)	AI (g)	Sample	BWi (kWh/t)
Tra 1-Ore B	10.3	15.2	0.033	Tra 1-Ore D	11.0*
Tra 1-Ore C	10.0	12.0	0.079	Tra 2-Waste A	10.0*
Tra 2-Ore A	9.1	13.3	-	Tra 3-Ore D	15.6*
Tra 3-Ore A	15.0	16.5	0.386	Tra 3-Ore E	12.0*
Tra 3-Ore B	-	8.4	0.025	BX 1-Waste	9.8*
Tra 3-Ore C	11.8	12.8	-	BX 2-Waste A	17.2*
BX 2-Ore A	9.6	11.9	0.063	BX 2-Waste B	12.7*
BX 2-Ore B	-	16.2	-	BX 2-Ore C	12.4*
BX 3-Ore A	-	11.5	0.033	Msed 3-Waste	6.1*
Por 3-Ore A	13.1	15.4	0.141	Por 3-Ore C	16.8*
Por 3-Ore B	-	17.9	0.172	Por 3-Waste	16.7*
				Var 10	13.5*
				Var 11	12.7*
				Var 12	16.4*

Note:
* Test done on assay rejects material and was finer than standard Bond ball mill feed. All BWi tests done with a closing screen size of 150 υm

IsaMill™ horizontal stirred milling is one technology used for ultrafine grinding, which is based on a high-intensity attrition mechanism. The relationship between the product size and energy input remains constant during scale-up from the bench-scale testing unit. A log-log plot of cumulative specific energy versus product size (‘signature plot’) is used to support the installed power requirements for a full-scale IsaMill™ (see Figure 13-4).

Figure 13-4: IsaMill Signature Plot (Two Stages, 4.5 mm & 2.5 mm)



Source: SRK, 2021

The flotation concentrate sample from test F-39 was submitted for IsaMill™ testing with a target product P_{80} size of 15 µm. Grinding was completed in two stages. Stage 1 used 4.5 mm grinding media with an intermediate target P80 of ~30 µm while Stage 2 used 2.5 mm grinding media to reach the final target P_{80} size. The results are plotted in Figure 13-4. The energy required to grind the sample from an F_{80} of 173 µm to the target product size was measured at 46.9 kWh/t. As the concentrate is expected to have a P_{80} size of 100 µm, the specific energy requirements were reduced to 35.8 kWh/t.

The 2019 PEA update report included results from an Eliason regrind test done on a sample of rougher concentrate. For a target P_{80} size of 17 µm, the specific energy requirement was estimated to be 41 kWh/t.

13.2.5 Gravity Recovery

Early testwork and recent mineralogy showed there was minimal free gold and gravity concentration tests resulted in limited gravity recoverable gold. Therefore, additional gravity recovery tests were not conducted as part of the 2020 program.

13.2.6 Whole Ore Leaching

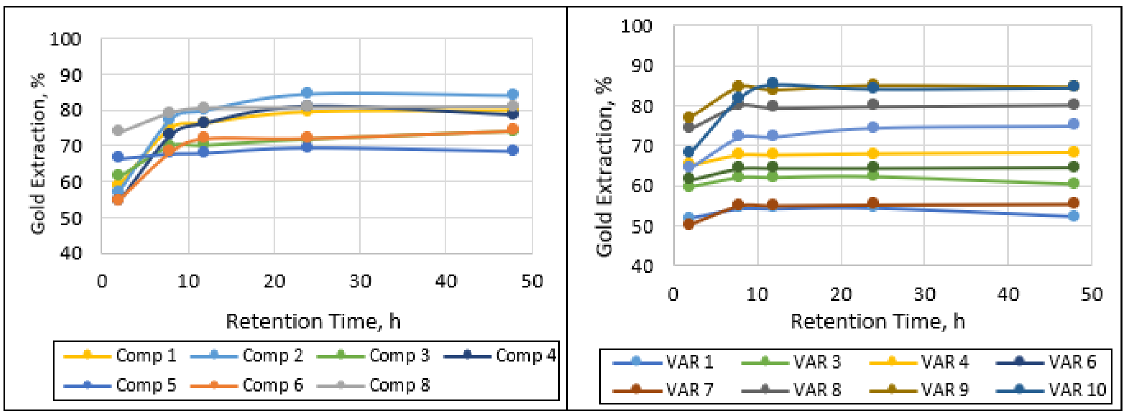
Whole ore leach tests were conducted on composite and variability samples from both Phase 1 and 2. These would then provide benchmark results to compare the float + concentrate/tailings leaching test results and allow comparison of the two prospective flowsheets.

Relatively aggressive leach conditions were employed to investigate the full potential of a whole ore leach flowsheet. Typical leach conditions to attack certain forms of tellurides include high pH, high dissolved oxygen (DO) and fine grind. Conditions included:

- 40 weight % solids
- pH of >12 with lime addition
- 6 hour pre-aeration stage
- DO of 20 ppm with oxygen addition
- NaCN concentration of 2 g/L maintained

Whole ore leach kinetics for composite samples (target P_{80} size of 60 µm) are presented in Figure 13-5.

Figure 13-5: Whole Ore Leach Kinetics – Composite Samples & Variability Samples

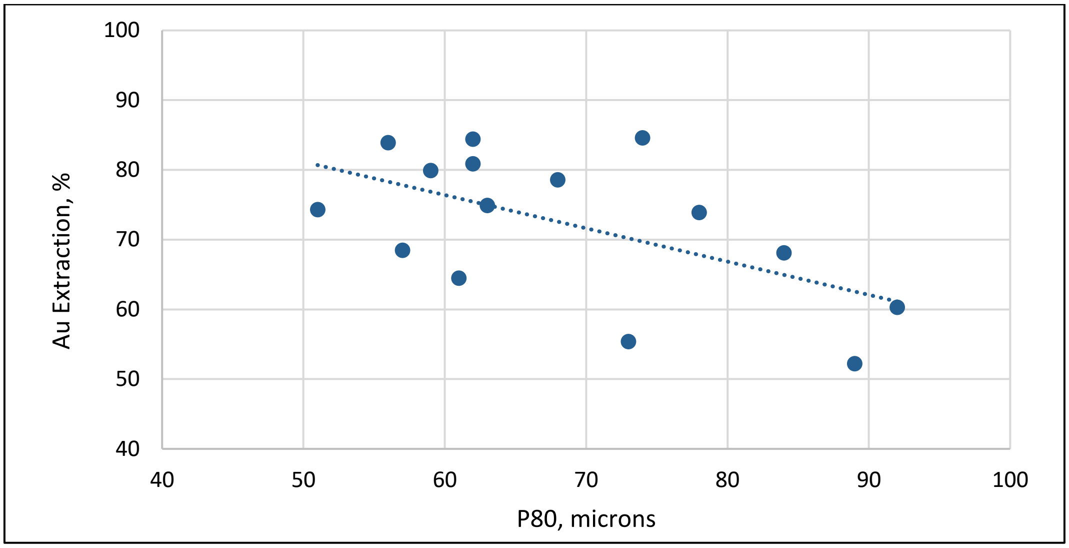


Source: SRK, 2021

In all cases the leach was complete within 24 hours; however, the gold leach extraction ranged from 72 to 84%. Whole ore leach kinetics for variability samples (target P_{80} size of 75 µm) are also shown as largely complete after 12 hours. For composite samples, gold leach extraction ranged from 72 to 84% and for variability samples, from 52 to 85%.

Whole ore leach grind sensitivity is presented in Figure 13-6 for all composites and variability samples. Gold extraction increases with decreasing grind P_{80} size; however, significant variability exists due to variable head grades and complex mineralogy.

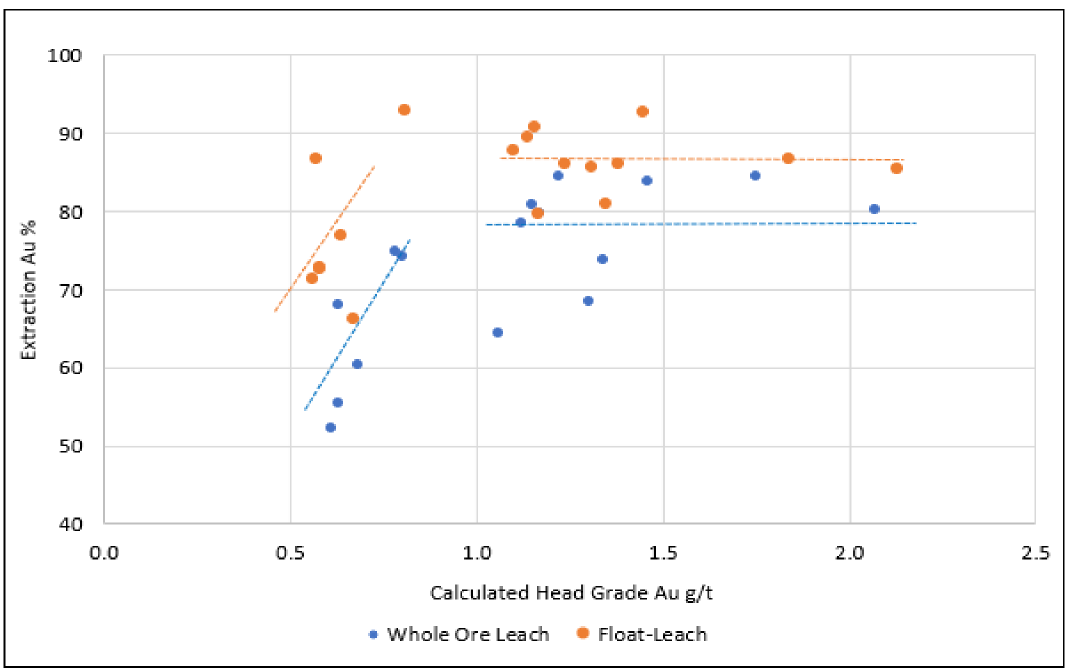
Figure 13-6: Whole Ore Leach Grind Size Sensitivity



Source: SRK, 2021

The test results for flotation, concentrate leaching and tailings leaching are discussed below. As a comparison of the two flowsheets being evaluated, Figure 13-7 shows the overall gold extractions achieved for the composite and variability samples tested.

Figure 13-7: Comparison of Float Leach Extraction versus Whole Ore Leach Extraction



Source: SRK, 2021

Both flowsheets showed lower overall extractions for low grade samples (0.7 g/t Au or lower) and relatively constant extractions from 1.0 g/t to 2.2 g/t Au. For the dashed lines shown in Figure 13-7, the float + concentrate/tailings leaching flowsheet indicated a 9% higher gold extraction for samples at 1.0 g/t or above.

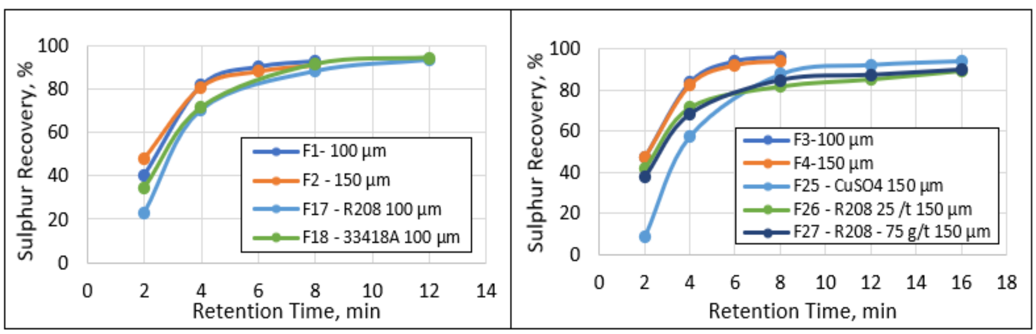
The following sub-sections cover flotation, concentrate leaching and tailings leaching which makes up the separation components of the PFS recommended flowsheet.

13.2.7 Flotation

Rougher flotation was investigated in Phase 1 using the four composite samples listed in Table 13-2, with the conditions including primary grind P_{80} size and collector addition. The base case scenarios used MIBC as a frother and PAX as a collector. No benefit was realized from the addition of R208, 3418A or $CuSO_4$.

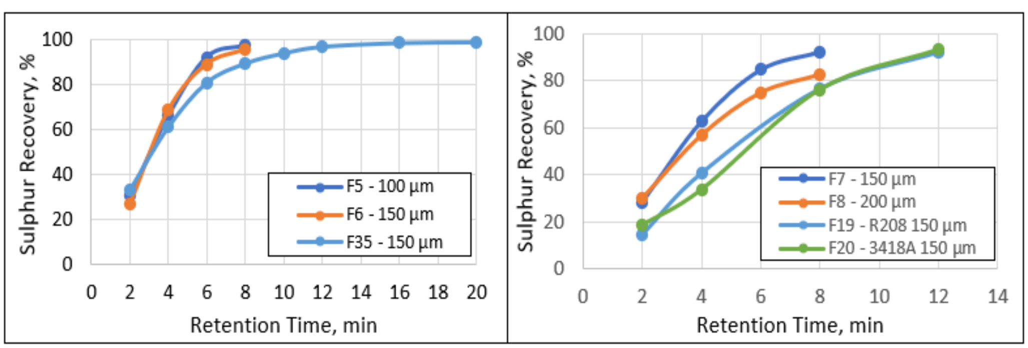
High sulphur recovery was achieved for Composites 1 to 4 at a primary grind P_{80} size of 100 to 150 µm. Composite 4 was tested at a coarser P_{80} size of 200 µm and showed 10% reduction in ultimate sulphur recovery between tests F7 and F8 (see Figure 13-8 and Figure 13-9).

Figure 13-8: Sulphur Recovery – Composite 1 & 2 Kinetic Series Rougher Flotation



Source: SRK, 2021

Figure 13-9: Sulphur Recovery – Composite 3 & 4 Kinetic Series Rougher Flotation

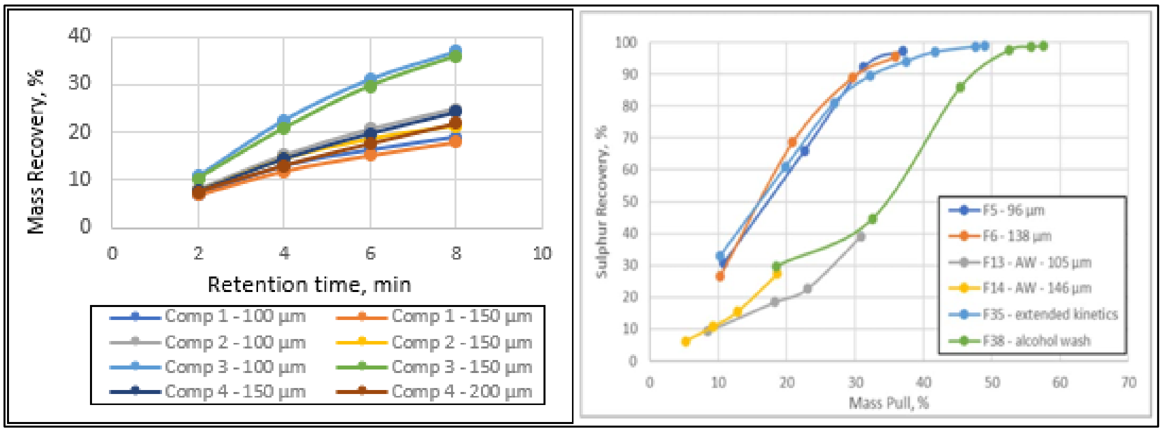


Source: SRK, 2021

Composite 3 showed a much higher mass pull compared with the remaining composites – almost double after eight minutes of float time (see Figure 13-10).

Excessive foaming was observed by the lab during testwork, attributed to the drilling compound used to aid drill core recovery. Drilling compound was added due to the friable nature of some of the core (see Figure 13-11). Various alcohol, acetone and water washes were conducted on the original sample to reduce this effect. This was also noted during the previous testwork programs conducted on the same drilling campaign samples. The acetone (test results labelled 'AW') and alcohol washes consistently reduced sulphur and gold recovery.

Figure 13-10: Mass Recovery – All Phase 1 Composites & Composite 3 Washing Trials



Source: SRK, 2021

Figure 13-11: Example of Crumbly and Competent Drill Core

Source: SRK, 2021

The issue of excessive foaming was observed in numerous samples of both Phase 1 and Phase 2 testing programs. This suggested it was not a core aging issue. The unusual froth properties resulted in higher mass pulls than targeted, but the differences were not consistent across all samples. In general, the use of acetone or alcohol pre-wash produced a more stable and consistent froth; however, gold and sulphur recoveries were negatively affected.

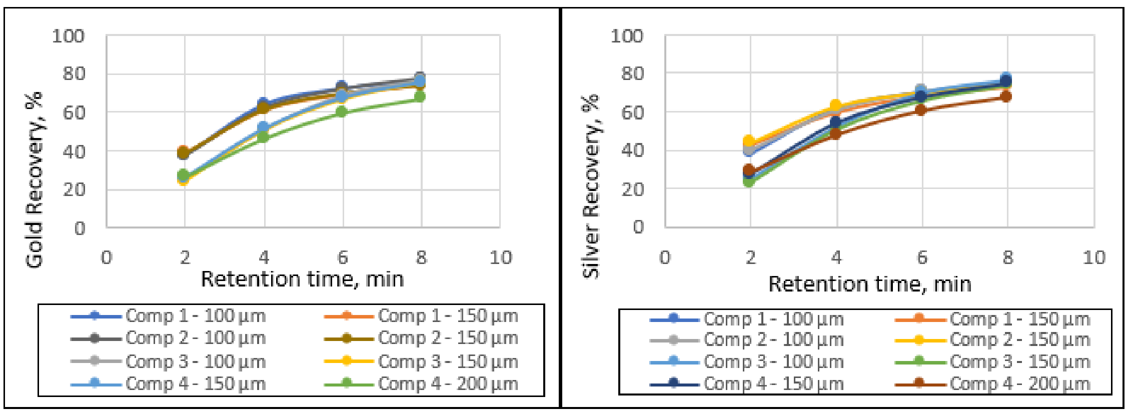
Changes to the drilling mud additive have been recommended and it is expected that future drill core samples will not be affected by these poor froth/foaming issues.

For this study, the unwashed flotation test results were used for estimating plant performance and the process plant design criteria discussed in Section 17. It remains an opportunity to reduce mass pull once additional core samples are collected and tested that do not exhibit these foaming characteristics. It is recommended to investigate this issue as part of future testwork programs.

For Composites 1 to 4, gold recovery ranged from 72 to 78% with a primary grind P_{80} size of 150 µm after eight minutes of float time. Silver recoveries were very similar to gold (see Figure 13-12).

With high sulphur recovery and relatively low gold recovery to the concentrate, leaching of the flotation tails is required to achieve improved gold recovery.

Figure 13-12: Gold & Silver Recovery – All Phase 1 Composites - Kinetic Series Rougher Flotation



Source: SRK, 2021

Optimized conditions for subsequent testing include:

- primary grind P_{80} 150 µm
- natural pH
- PAX collector addition of 110-130 g/t
- MIBC frother addition as required

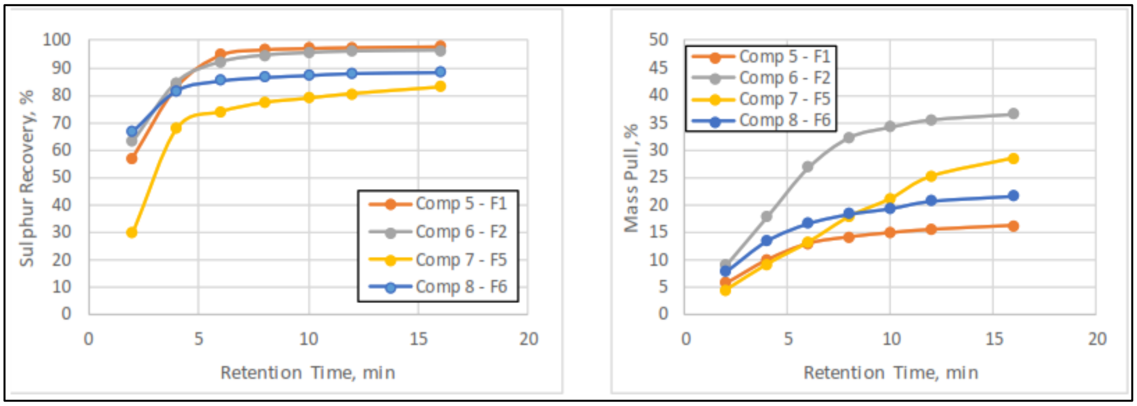
X-ray diffraction (XRD) was conducted on Composites 1 to 3 rougher concentrates to determine the non-sulphide gangue reporting to concentrate. In all cases, the main non-sulphide gangue reporting to rougher concentrate was a range of feldspars and micas with minor kaolinite.

Phase 2 rougher flotation tests were performed under conditions developed in Phase 1 testing on fresh drill core samples. (See Table 13-3 for list of Phase 2 composites.)

Composites 5 and 6 showed high sulphur recovery (>95%). However, Composite 6 was foaming and resulting in increased mass pull with expected entrainment of non-sulphide gangue. Composites 7 and 8 showed sulphur recovery ranging from 80 to 88% (see Figure 13-13 and Figure 13-14).

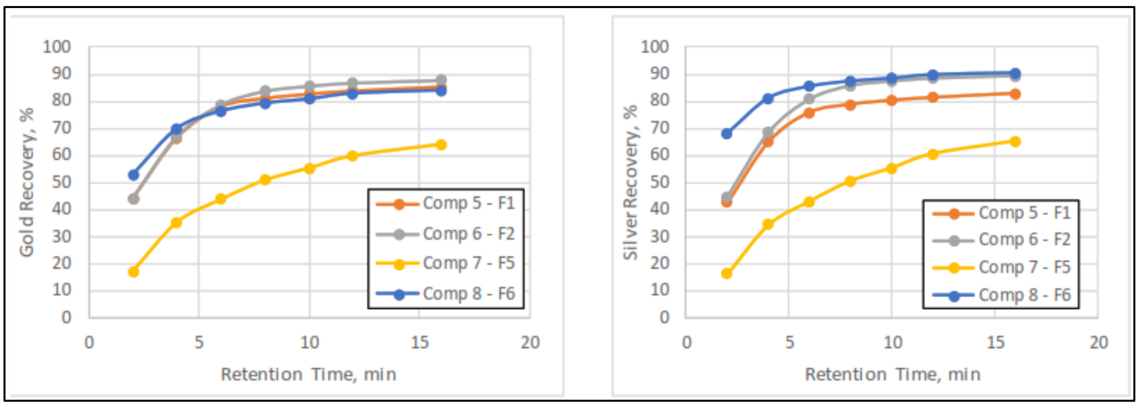
Composites 5, 6 and 8 reported relatively high gold recovery compared with Phase 1 composites – ranging from 78 to 82% after eight minutes. Composite 7, with low sulphide and gold head grades (0.14% S, 0.8 g/t Au), showed reduced gold and silver recovery.

Figure 13-13: Sulphur Recovery and Mass Pull – Phase 2 Composites - Kinetic Series Rougher Flotation



Source: SRK, 2021

Figure 13-14: Gold & Silver Recovery – Phase 2 Composites - Kinetic Series Rougher Flotation



Source: SRK, 2021

A series of cleaner flotation tests were conducted at a primary P_{80} size of 150 µm to optimize gold recovery. Tests F21 to F24 were conducted in 10 kg batches to generate sufficient sample for downstream testing, with a mass pull between 11 and 23%. Composite 3 (Test F24) exhibited excess foaming along with higher mass pull (32%). Gold recovery ranged from 66 to 75% and sulphur recovery from 92 to 93% (see Table 13-14).

Tests F31 to F34 were conducted on 2 to 4 kg batches to generate sample to optimize concentrate regrind size for cyanide leaching. Mass pulls were between 13 and 22% with gold recovery ranging from 55 to 72% and sulphur recovery between 75 and 92%.

Phase 2 composites showed similar recoveries to Phase 1 composites: 61 to 76% for gold and 85 to 95% for sulphur; albeit at lower mass pulls between 12 and 14%.

Phase 2 variability samples showed similar recoveries to Phase 1/2 composites: 63 to 84% for gold and 84 to 98% for sulphur, with mass pulls ranging from 2 to 23% (see Table 13-15).

Table 13-14: Summary of Phase 1 Composite Cleaning Tests

Sample	Test	Overall Mass Pull %	Concentrate Grade			Recovery %		
			Au g/t	Ag g/t	S %	Au	Ag	S
Comp 1	F21B	11	7.41	44.4	23.2	70.0	69.3	92.6
	F31	15	6.50	46.9	21.8	72.2	76.6	91.8
Comp 2	F23	23	4.41	21.7	12.1	74.9	64.5	92.7
	F32	13	7.24	35.2	24.6	66.3	73.6	88.7
Comp 3	F24	32	2.85	15.4	17.1	73.3	70.0	93.1
	F33	22	3.23	19.2	19.2	54.7	55.1	75.4
Comp 4	F22	13	5.55	39.4	27.2	65.8	70.3	91.8
	F34	20	3.90	23.2	18.9	63.3	64.1	82.5

Table 13-15: Summary of Phase 2 Composite Cleaning Tests

Sample	Test	Overall Mass Pull %	Concentrate Grade			Recovery %		
			Au g/t	Ag g/t	S %	Au	Ag	S
Comp 5	F7	11.9	7.0	18.3	26.7	71.2	72.4	94.9
Comp 6	F8	12.8	3.2	19.5	15.0	61.3	68.8	87.3
Comp 8	F9	14.4	9.8	89.0	23.6	76.5	79.2	84.6
Var 1	F10	6.2	29.4	33.0	65.0	62.8	89.5	94.1
Var 3	F11	7.7	6.2	19.4	28.1	74.2	59.3	90.4
Var 4	F12	11.0	3.5	14.0	35.0	67.7	64.0	92.9
Var 6	F17	8.4	13.1	107	38.8	77.8	75.1	92.0
Var 7	F14	4.3	8.4	44.8	30.7	62.8	60.2	84.1
Var 8	F15	10.3	16.0	112	32.5	77.4	72.9	94.9
Var 9	F16	1.8	60.5	857	26.6	75.8	79.7	84.4
Var 10	F18	4.4	15.6	6.5	19.9	83.6	36.9	92.6
Var 11	F22	23.4	3.88	7.0	27.6	82.8	74.7	97.5
Var 12	F23	23.1	2.79	4.5	21.5	83.1	70.2	94.9

13.2.8 Flotation Concentrate Leaching

Phase 1 composite cleaner concentrates were cyanide leached under the conditions below, with results summarized in Table 13-16, including the varying grind P_{80} size.

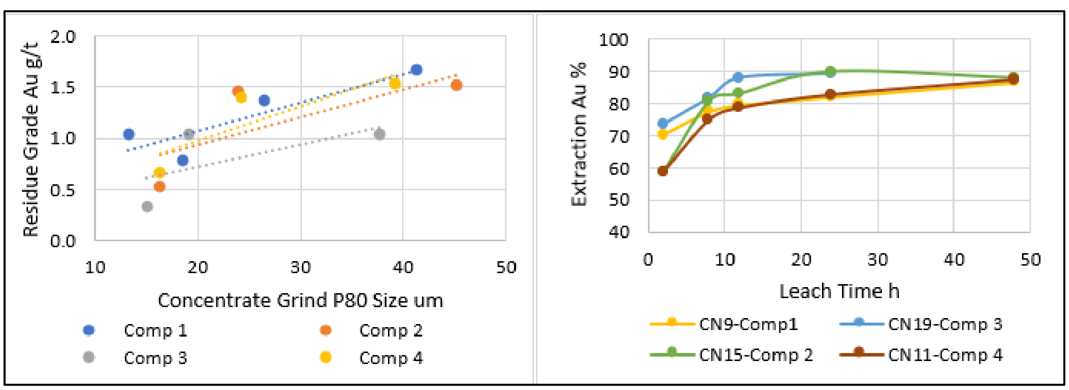
- pulp density of 40 weight % solids
- pH of 11 to 11.5 with lime addition
- DO of 20 ppm with oxygen addition
- NaCN concentration of 2 g/L maintained

Table 13-16: Phase 1 Flotation Concentrate Leach at Varying Grind Size

Sample	Test	Grind Size, P80 µm	Residue Grade g/t	Au Extraction %	Ag Extraction %	NaCN Consumption kg/t conc	Lime (CaO) Consumption kg/t conc
Comp 1	CN9	13.2	1.04	86.3	99.1	4.0	6.2
	CN10	18.6	0.78	88.9	97.8	4.1	5.7
	CN21	26.5	1.36	76.7	85.1	2.8	4.3
	CN22	41.3	1.67	70.4	79.3	2.1	3.4
Comp 2	CN15	16.3	0.53	87.3	93.6	6.7	7.0
	CN16	23.9	1.47	77.7	89.4	3.3	3.5
	CN17	45.2	1.51	77.0	90.0	3.3	3.5
Comp 3	CN18	15.0	0.34	88.4	95.8	8.5	8.7
	CN19	19.2	1.04	69.8	91.1	3.0	3.3
	CN20	37.7	1.04	78.2	84.5	2.9	3.2
Comp 4	CN11	16.4	0.67	87.0	98.0	4.8	5.0
	CN23	24.3	1.39	61.7	75.1	3.2	3.5
	CN30	39.1	1.53	58.5	75.9	2.7	3.1

Residue gold grade by grind size and extraction by leach time is summarized in Figure 13-15. Leach extraction for Composites 2 and 3 was complete in 30 hours while Composites 1 and 4 were extended beyond 30 hours.

Figure 13-15: Phase 1 Composites - Concentrate Leach Grind Sensitivity and Leach Kinetics



Source: SRK, 2021

Based on the concentrate leach kinetics, 30 hours residence in the leach tanks is recommended as the residue slurry will be transferred to the flotation tails leach circuit for an additional 20 hours of residence time. These times were included in the process design criteria covered in Section 17.

Phase 2 flotation concentrate leach results are summarized in Figure 13-16. Phase 1 composite cleaner concentrates were cyanide leached under the conditions below, with results summarized in Table 13-17, including the varying grind P_{80} sizes:

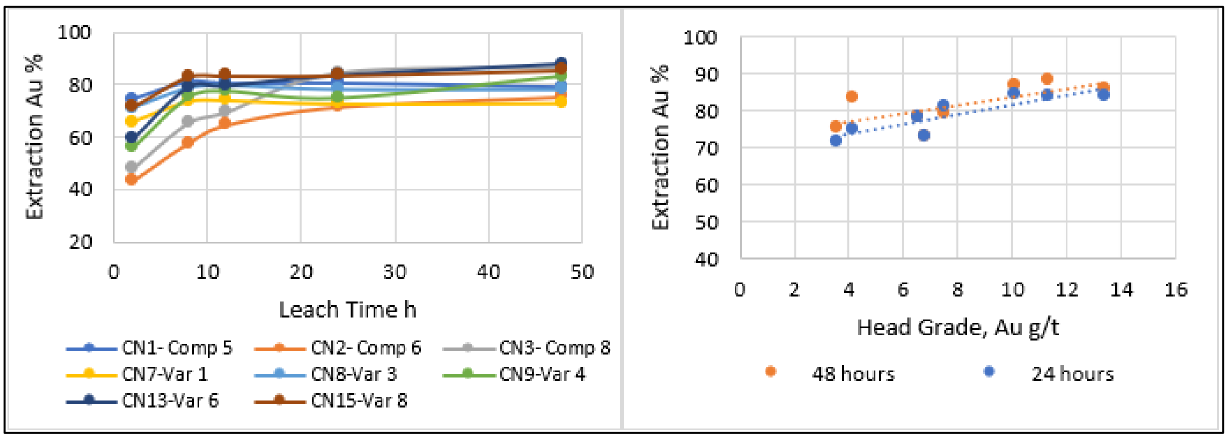
- pulp density of 40 weight % solids
- pH of 11 to 11.5 with lime addition
- DO of 20 ppm with oxygen addition
- NaCN concentration of 2 g/L maintained

At 48 hours, gold extraction ranged from 62 to 97% and silver extraction from 80 to 96%. Sodium cyanide consumption varied from 1.3 to 11.6 kg/t of concentrate and lime consumption was between 2.2 and 7.8 kg/t of concentrate.

Phase 2 samples similarly demonstrated most of the extraction was completed within 30 hours. Flotation Tailings Leaching

Figure 13-16 below presents concentrate leach kinetics and compares gold extraction versus head grade at 24 and 48 hours. For clarity, both graphs exclude test CN16 with a high concentrate grade of 60.5 g/t Au.

Figure 13-16: Phase 2 Samples - Concentrate Leach Kinetics and Extraction by Head Grade



Source: SRK, 2021

Table 13-17: Phase 2 Flotation Concentrate Leach – Regrind P80 size of 15 µm

Sample	Test	Residue Grade Au g/t	Au Extraction %	Ag Extraction %	NaCN Consumption kg/t conc	Lime (CaO) Consumption kg/t conc
Comp 5	CN1	1.56	79.2	86.6	3.6	4.7
Comp 6	CN2	0.89	75.2	86.6	2.2	3.7
Comp 8	CN3	1.35	86.7	88.3	4.6	2.2
Var 1	CN7	1.84	73.0	90.2	5.2	6.2
Var 3	CN8	1.45	78.0	90.5	3.9	3.6
Var 4	CN9	0.70	83.3	88.3	3.6	4.3
Var 6	CN34	0.93	91.8	96.1	1.3	5.8
Var 7	CN14	3.28	62.4	83.2	4.8	7.8
Var 8	CN15	1.92	85.6	95.1	5.1	3.9
Var 9	CN16	1.92	97.1	95.8	11.6	5.8
Var 10	CN35	0.68	95.9	88.7	3.9	6.8
Var 11	CN44	0.80	77.4	80.0	2.6	3.7
Var 12	CN45	0.51	84.1	82.4	1.1	3.6

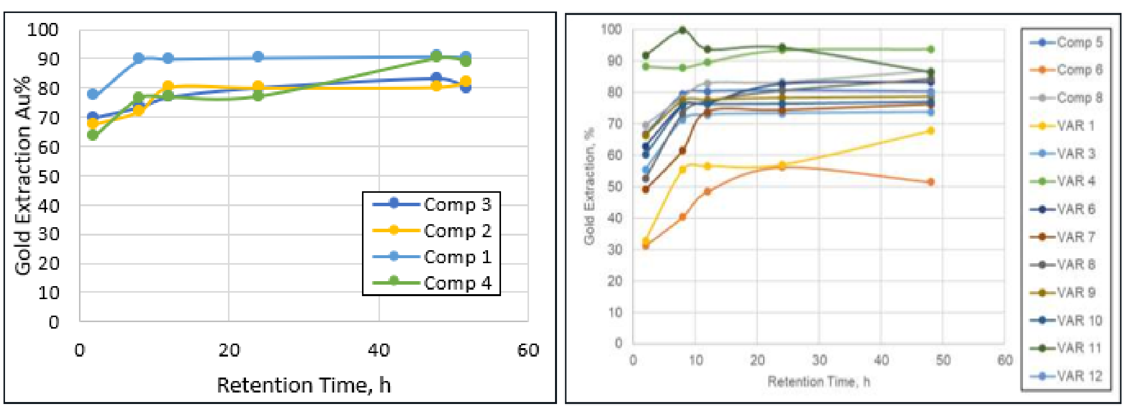
13.2.9 Flotation Tailings Leaching

Phase 1 composite flotation tailings samples were cyanide leached under the following conditions with the results shown in Figure 13-17 below and summarized in Table 13-18:

- pulp density of 40 weight % solids
- pH of 11 to 11.5 with lime addition
- DO of 20 ppm with oxygen addition

Gold extractions were generally complete within 12 hours. A total tailings leach adsorption time of 21 hours was nominated for design.

Figure 13-17: Tailings Leach Kinetics – Phase 1 and Phase 2 Samples



Source: SRK, 2021

Table 13-18: Phase 2 Flotation Tailing Leach

Sample	Test	Residue Grade Au g/t	Au Extraction %	Ag Extraction %	NaCN Consumption kg/t plant feed	Lime (CaO) Consumption kg/t conc
Comp 5	CN4	0.08	80.2	73.5	0.33	0.95
Comp 6	CN5	0.13	51.5	76.2	0.30	0.62
Comp 8	CN6	0.07	86.8	83.2	0.33	0.74
Var 1	CN10	0.06	67.8	64.3	0.26	0.72
Var 3	CN11	0.04	73.7	59.5	0.27	0.75
Var 4	CN12	0.04	93.8	75.0	0.68	0.77
Var 6	CN17	0.19	70.0	84.9	0.21	1.02
Var 6[1]	CN39	0.06	83.2	80.1	0.20	0.81
Var 7	CN18	0.09	89.8	89.2	0.18	0.77
Var 8	CN19	0.09	84.3	90.5	0.19	0.75
Var 9	CN20	0.08	78.7	81.9	0.25	0.85
Var 10	CN40	0.03	77.0	26.4	0.19	0.72
Var 11	CN46	0.03	86.4	58.6	0.13	0.90
Var 12	CN47	0.04	79.8	67.2	0.15	1.30

Var 6 was repeated

For the Phase 2 tailings samples, low residue grades were generally attained while Variability 6 test (CN17) was repeated as test CN39 and returned a much lower residue grade.

Sodium cyanide consumption ranged between 0.18 to 0.68 kg/t plant feed while lime consumption was from 0.62 to 1.3 kg/t plant feed.

13.2.10 Bulk Flotation-Leach

A bulk flotation test was conducted to generate sufficient sample for cyanide detoxification, thickening, rheology and filtration. The bulk sample was comprised of Phase 1 composites, as presented in Table 13-19.

Table 13-19: Bulk Composite Composition

Sample	Composition %
Comp 1	27
Comp 2	31
Comp 3	21
Comp 4	21

Bulk flotation test results are shown in Table 13-20 with extractions and overall recoveries summarized in Table 13-21.

The bulk flotation concentrate was reground to a P_{80} size of 15 µm and leached according to optimized conditions. The concentrate leach residue was transferred to flotation tails leach, as per proposed flowsheet conditions. The leach density was increased to 48 weight % solids, without adverse effects.

[1] Repeat test

Table 13-20: Bulk Flotation Test Results

Product	Wt	Assays			Distribution %		
	%	Au, g/t	Ag, g/t	S, %	Au	Ag	S
1st Cleaner Concentrate	18.8	5.14	29.3	19.4	73.8	77.5	96.4
1st Cleaner Tailings	10.2	0.93	4.6	0.35	7.2	6.6	0.9
Rougher Tailings	71.0	0.35	1.6	0.14	19.0	16.0	2.6
Head (Calc.)	100.0	1.31	7.1	3.78	100.0	100.0	100.0
Head (Direct.)		1.22	6.4	3.82			
Combined Products	**Wt**	**Assays**			**Distribution %**		
	%	**Au, g/t**	**Ag, g/t**	**S, %**	**Au**	**Ag**	**S**
1st Cleaner Concentrate	18.8	5.14	29.3	19.4	73.8	77.5	96.4
Rougher Concentrate	29.0	3.66	20.6	12.7	81.0	84.0	97.4
Rougher Tailings	71.0	0.35	1.6	0.14	19.0	16.0	2.6
Combined Tailings	81.2	0.42	2.0	0.17	26.2	22.5	3.6

Table 13-21: Bulk Flotation Leach Extraction and Overall Recovery

Parameter	Test	Au	Ag
Flotation feed calc head, g/t	CN-36, CN-37 avg	1.14	6.60
Flotation concentrate, calc head g/t	CN-35	4.25	22.80
Concentrate leach, extraction %	CN-35	77.8	87.30
Combined (conc residue + float tails residue, g/t)	CN-36, CN-37 avg	0.17	0.90
Overall Float-leach Recovery		**85.0**	**86.4**

13.2.11 Recovery Estimates for Recommended Flowsheet

The test results summarized in Table 13-22 were used to estimate overall plant recovery in the financial model. The selected tests were done close to the nominated primary grind P80 size of 150 µm, over the range of 130 to 160 µm. Rougher tailings sample sizing's were checked against mill discharge sample sizing's to confirm their validity.

Nominated mass pull to final concentrate is 15% in the design criteria with a target concentrate regrind P_{80} size of 15 µm. Test results ranged from a P_{80} of 12 to 24 µm. Cyanide leach tests were conducted for 48 hours and sampled at 12 and 24 hours. Plant design is based on 30 hours concentrate leach time and 20 hours flotation tails leach time. The overall leach extraction was downgraded by 1% to account for plant solution losses and shown as "Plant Recov" in Table 13-22.

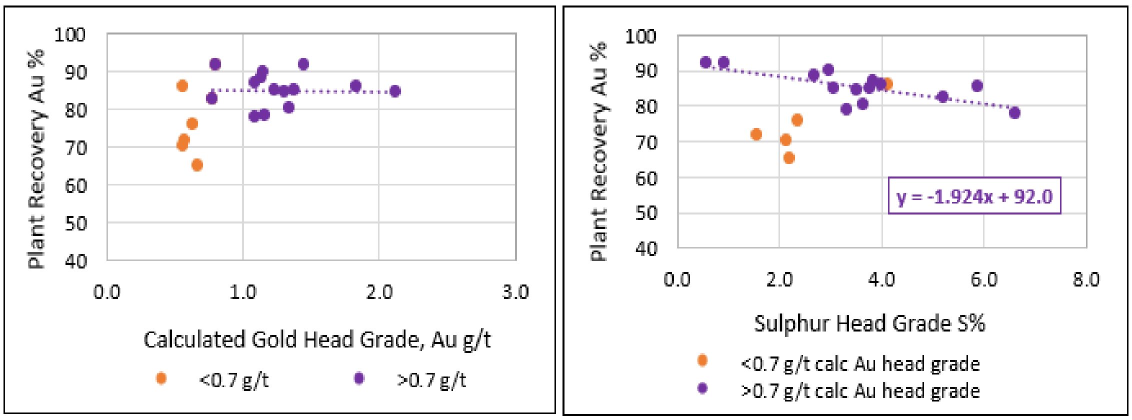
Table 13-22: Cyanide Leach Tests used for Float-Reg Grind-Leach Flowsheet for Financial Model[2]

Sample	Flotation Conc % Distribution		Flotation Conc CN % Distribution		Flotation Tail CN % Distribution		Overall Float Leach Recovery		Head Grade (calc from float test)			Overall Residue		Plant Recov	Main Litho
	Au %	Ag %	Au %	Ag %	Au %	Ag %	Au %	Ag %	Au g/t	Ag g/t	S %	Au g/t	Ag g/t	Au %	
Comp 1	70.0	69.3	62.2	67.8	27.1	20.0	89.3	87.7	1.14	6.0	2.7	0.11	0.53	88.3	Tuff/Por
Comp 2	74.9	64.5	65.4	60.4	20.6	31.2	86.0	91.6	1.38	5.1	3.07	0.19	0.90	85.0	Tuff/Por
Comp 3	73.3	70.0	64.8	67.1	21.3	27.4	86.1	94.5	1.24	4.9	5.88	0.20	0.85	85.1	Trachyte
Comp 4	65.8	70.3	57.2	68.9	30.6	20.6	87.8	89.4	1.10	5.1	3.86	0.12	0.51	86.6	Trachyte
Comp 5	71.2	72.4	26.4	62.7	23.1	20.3	79.5	83.0	1.17	3.0	3.34	0.26	0.85	78.5	Trachyte
Comp 6	61.3	68.8	46.1	59.6	19.9	23.8	66.0	83.4	0.67	3.6	2.21	0.23	0.87	65.0	Trachyte
Comp 8	76.5	79.2	66.3	69.9	20.4	17.3	86.7	87.2	1.84	16.1	4.01	0.25	2.30	85.7	Porphyry
Var 1	65.0	62.8	47.5	56.6	23.7	23.9	71.2	80.6	0.56	2.7	2.16	0.17	0.64	70.2	Trachyte
Var 3	74.2	59.3	57.9	53.7	19.0	24.2	76.9	77.9	0.64	2.5	2.38	0.15	0.62	75.9	Trachyte
Var 4	67.7	64.0	56.4	56.5	30.3	27.0	86.7	83.5	0.57	2.4	4.14	0.11	0.65	85.7	Porphyry
Var 6	60.9	61.1	53.6	57.7	27.4	33.0	81.0	90.8	1.35	12.6	3.65	0.28	1.34	80.0	Breccia
Var 7	62.8	60.2	39.2	50.1	33.4	35.5	72.6	85.6	0.58	3.2	1.57	0.23	0.83	71.6	Trachyte
Var 8	77.4	72.9	66.3	69.3	19.1	24.5	85.3	93.9	2.13	15.8	3.53	0.28	0.98	84.3	Trachyte
Var 9	75.8	79.7	73.6	76.4	19.0	16.6	92.6	93.0	1.45	19.5	0.57	0.11	1.42	91.6	Breccia
Var 10	83.6	36.9	80.2	32.7	12.6	16.7	92.8	49.4	0.81	0.8	0.93	0.06	0.54	91.8	Andesite
Var 11	82.8	74.7	64.1	59.8	14.9	14.8	78.9	74.6	1.10	2.2	6.63	0.22	8.00	77.9	Meta Sed
Var 12	83.1	70.2	69.9	57.8	13.5	20.0	83.4	77.9	0.78	1.5	5.23	0.15	0.82	82.4	Breccia
PEA Comp	-	-	-	-	-	-	90.7	95.1	1.16	6.0	2.98	0.11	0.29	89.7	Master

The relationship of overall plant recovery versus calculated gold head grade provided very little correlation. Similarly, relationships between residue grades and lithology also were associated. To provide an estimate of plant recovery, Figure 13-18 shows a reasonable relationship with sulphur head grade; however, this only applied to samples with gold head grades >0.7 g/t. For lower grade samples, a flat recovery value of 73% was recommended.

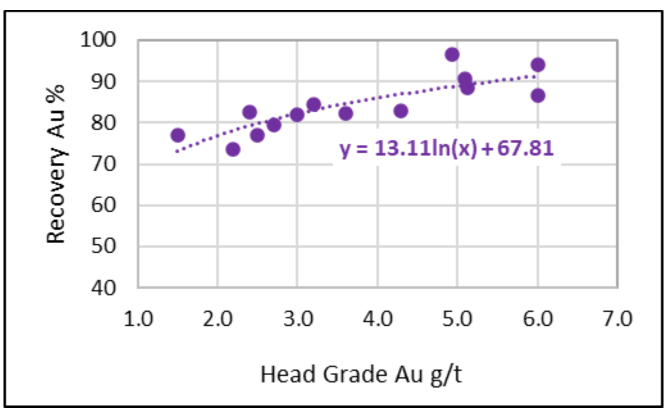
Figure 13-19 shows the silver plant recovery versus head grade relationship applied in the financial model.

Figure 13-18: Plant Recovery vs Gold and Sulphur Head Grade



Source: SRK, 2021

Figure 13-19: Plant Recovery vs. Silver Head Grade



Source: SRK, 2021

13.3 Cyanide Detoxification

Both batch and continuous cyanide detoxification tests were conducted on combined concentrate leach and tailings leach samples. Batch tests were done on products from tests F21B and F22 while continuous testing was done on bulk test (F39) combined leach residues.

Effective cyanide detoxification was demonstrated under favourable reagent consumptions in the presence of elevated residual cyanide levels.

Results of the continuous testing include the following (see Table 13-23):

- weakly acid dissociable (WAD) cyanide was effectively reduced from 861 mg/L to less than 1 mg/L after 89 minutes; measured by both picric acid and distillation methods
- total cyanide was reduced to less than 5 mg/L
- an SO_2 requirement of 4.69 g/g CN_{wad} is typical of air/SO_2 process using sodium metabisulphite (SMBS)

Table 13-23: Cyanide Detoxification Test Results – Optimized Continuous Test

Feed CNwad	Feed CN total	Retention Time	Discharge CN WAD		Discharge CN total	Ratio SO2 / CNwad	Ratio Cu / CNwad	Ratio lime / CNwad
mg/L	mg/L	min	mg/L	mg/L picric	mg/L	g/g	g/g	g/g
861	900	89	<0.1	0.9	4.65	4.69	0.10	2.58

CN_{wad} measured by picric acid method

13.4 Solid/Liquid Separation

Solid/liquid separation testwork was performed on bulk samples of flotation tailings and detoxified final leach tailings. Both static and dynamic settling tests were conducted on both samples, with further filtration tests conducted on the detoxified tailings sample.

13.4.1 Flotation Tailings

The flotation tailings sample is composed of rougher tails and 1st cleaner tails from bulk test F39. The sample particle P_{80} size was measured at 132 µm with 46% finer than 20 µm.

Static settling tests were performed for flocculant screening and establishing preliminary settling test conditions using both Magnafloc 1687 and 155. The results indicate the sample responded well to Magnafloc 155, a high molecular weight anionic flocculant. At a diluted feed solids density of 5%, the static settling rate of tailings required a unit area of 0.19 m^2/tpd. The resulting underflow contained 57% solids by weight while the overflow total suspended solids (TSS) was 81 mg/L.

Dynamic settling tests were conducted to verify static settling results and to determine the commercial thickener sizing parameters. At a unit area of 0.22 m^2/tpd, it was reported that a sequential dosage of BASF Magnafloc 1687 coagulant and Magnafloc 155 are required to produce a similar overflow quality. The results are summarized in Table 13-24.

Table 13-24: Effect of Flocculant Dosage – Flotation Tailings

Unit Area (m2/tpd)	Mag 1687 (g/t)	Mag 155 (g/t)	Overflow TSS (mg/L)
0.22	-	60	272
0.22	-	70	165
0.22	-	80	150
0.22	-	90	120
0.22	20	15	80
0.22	20	20	64

Initial dynamic thickening tests were conducted at a dosage rate of 20 g/t for both the selected coagulant and flocculant. The thickener underflow solids density was measured at 60.9% after a residence time of 2.2 hours with a similar overflow TSS level.

Additional dynamic thickening tests were requested to generate a target underflow solids density at 50% by weight. The results are shown in Table 13-25. A thickener underflow density of 51% is achieved at a unit area of 0.05 m^2/tpd; however, the resulting overflow TSS is 279 mg/L that indicates additional polymer(s) dosage is required if a lower TSS is desired.

Table 13-25: Dynamic Settling Results - Additional Testing of Flotation Tailings

Mag 1687 (g/t)	Mag 155 (g/t)	Unit Area (m^2/tpd)	Settling Rate (t/h/m^2)	Underflow Solids Density (wt.%)	Overflow TSS (mg/L)	Residence Time (h)	Underflow Yield Stress (Pa)
30	30	0.10	0.42	55.8	63	1.0	62
30	30	0.08	0.52	53.2	110	0.8	35
30	30	0.06	0.69	51.4	166	0.6	18
30	30	0.05	0.83	50.8	279	0.5	33

For process design, a settling rate of 0.52 t/h/m^2 was assumed. This resulted in a flotation tailings high-rate thickener diameter of 54 m.

13.4.2 Detoxified Final Tailings

Thickening tests including static and dynamic settling and rheology of the thickened underflow were performed on the detoxified tailings sample. Detoxified tailings consist of flotation tailings leach residue including reground concentrate leach residue. The cyanidation detoxification discharge sample has a particle P_{80} size of 97 µm.

The cyanidation detoxification discharge sample also responded well to Magnafloc 155. At a diluted feed density of 7.2% by weight, the tailings required a unit area of 0.18 m^2/tpd. The resulting underflow contained 55% solids by weight while the overflow TSS was measured at 14 mg/L.

It was noted that a sequential dosage of BASF Magnafloc 1687 coagulant and Magnafloc 155 are required under the dynamic settling test conditions to produce a similar overflow quality. This can be seen from the test results presented in Table 13-26.

Table 13-26: Effect of Flocculant Dosage – Final Tailings

Unit Area (m2/tpd)	Mag 1687 (g/t)	Mag 155 (g/t)	Overflow TSS (mg/L)
0.22	-	40	396
0.22	20	20	279
0.22	20	30	154
0.17	20	40	85
0.17	30	40	53

The dynamic thickening tests results are presented in Table 13-27. The highest solids density of the thickener underflow is 56% with a required unit area of 0.17 m^2/tpd. The resulting TSS level in the thickener overflow was measured at 53 mg/L. When extending the settling process by another 30 minutes, the underflow solids content was slightly increased from 56% to 57%.

Table 13-27: Dynamic Settling Results – Final Tailings

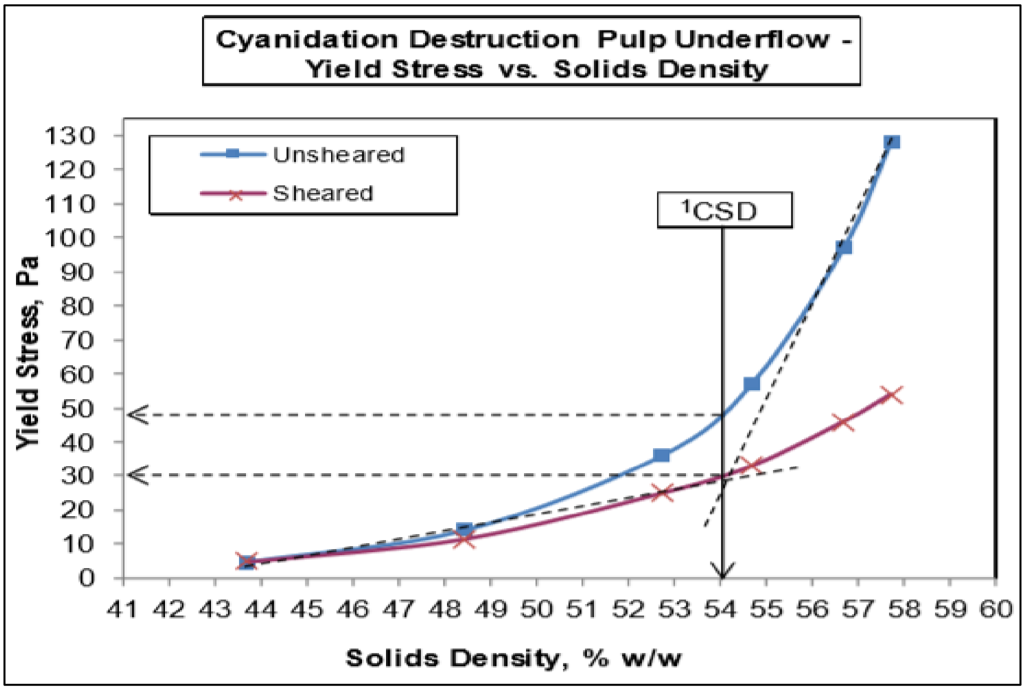
Mag 1687 (g/t)	Mag 155 (g/t)	Unit Area (m2/tpd)	Settling Rate (t/h/m2)	Underflow Solids Density (wt.%)	Overflow TSS (mg/L)	Residence Time (h)	Underflow Yield Stress (Pa)
30	40	0.17	0.25	56.2	53	1.7	96
30	40	0.14	0.30	55.4	48	1.4	82
30	40	0.11	0.38	54.4	44	1.1	86
30	40	0.08	0.52	52.9	129	0.7	78
30	40	0.05	0.83	47.1	230	0.5	32

For process design, a settling rate of 0.25 t/h/m^2 was assumed. This resulted in a flotation tailings high-rate thickener diameter of 68 m.

Underflow rheology tests were conducted to assess the pulp inter-particle interaction; that is, the deviation of actual specific gravity at varied % solids content to the specific gravity of dry solids.

The test data are used to assess the flowability with the underflow sample exhibiting insignificant inter-particle interactions. The critical solids density (CSD) of the underflow sample is about 54% corresponding to a yield stress of 48 Pa under unsheared conditions. Plug flow response was observed in this situation. For the sheared condition, the underflow sample behaved “flow-friendly” (or thixotropic) at a solid density of up to 58%. The results are presented in Figure 13-20.

Figure 13-20 Yield Stress vs Solids Content – Tailings Thickener Underflow



Source: SRK, 2021

13.4.3 Filtration of Final Tailings

Initial filtration tests were carried out by SGS to assess the performance of both vacuum and pressure filtration technology on the thickened detoxified tailings sample from the bulk test. Vacuum filtration was conducted at 0.68 bar vacuum level, while as pressure filtration was conducted at 6.9 bar pressure level. The test results presented in Table 13-28 show that pressure filtration generates a lower cake moisture and higher filtration rate when compared with vacuum filtration. The cake moisture achieved by pressure filtration is between 19 and 20%.

Table 13-28: Vacuum and Pressure Filtration Test Results (55% Solids Feed)

Filtration Method/ Pressure	Filter Cloth	Form + Dry Time (min)	Cake Thickness (mm)	Cake Moisture (w/w%)	Filtration Rate* (kg/m²h)
Vacuum @0.68 bar	Testori P6583 TC	15.1	15	26.6	86
		7.8	10	25.0	113
		9.8	10	24.5	90
		13.3	10	23.8	67
		17.2	10	23.3	51
		13.8	5	21.6	34
		33.8	15	23.2	39
Pressure @6.9 bar	Testori P4408 TC	3.0	10	19.2	68
		5.0	15	19.2	89
		9.0	20	19.3	94
		13.8	25	20.1	95
Pressure @9.9 bar	Testori P4408 TC	2.5	10	19.4	70
		5.6	15	19.0	86
		8.7	20	18.5	96
		12.4	25	18.8	100

Notes:
* Vacuum filtration rate is based on form and dry times only; pressure filtration rate is based on full cycle time including form and dry times, plus 10 min technical service time (feeding, discharging, and cloth washing).

A further filtration test was completed by Outotec using membrane squeeze with a target moisture between 15 and 16% on one detoxified tailings sample. The tested sample contained 53% solids, similar to the expected solids density from the thickener underflow. The particle P80 size was measured at 113 µm with 50% particles below 20 µm.

A total of five tests were completed on the Larox 100 bench unit using AITE S400 filter cloth. The results (Table 13-29) indicate that the cake moisture 15% could be achieved with membrane squeeze and air-drying. No cloth blinding was observed when releasing cake from the filter cloth. It is recommended that additional variability testwork be conducted during the next study phase.

Table 13-29: Pressure Filtration Test Results Outotec (with Membrane Squeeze)

Test #	Cycle Time (min)	Pumping (bar)	Pressing (bar)	Drying (bar)	Cake Thickness (mm)	Cake Moisture (w/w%)	Filtration Rate (kg/m^2.h)
T1-50	12	6	12	8	33	19.4	129
T2-40	12	6	12	10	30	15.4	116
T3-40	11	6	12	10	31	15.4	120
T4-40	10	6	12	8	27	17.1	116
T5-40	11	7	12	10	28	15.2	111

The current plan is for filtered tailings to be transported to the WSF using the same haul truck fleet that is delivering feed to the plant. Tailings will be co-disposed with waste rock material at the WSF.

13.5 Conclusions

During 2020, First Mining completed a comprehensive comminution and metallurgical testwork program to support the PFS. This included head grade analyses, mineralogy, a full suite of comminution, flotation, and leach tests; cyanide detoxification, rheology, and solid/liquid separation. Testwork was conducted by SGS Lakefield, Canada in two phases: Phase 1 used available coarse reject material from the 2016 drilling campaign and Phase 2 used fresh HQ drill core from the 2020 winter drilling campaign.

Tests were performed on mineralization that is considered to be representative of plant feed, based on a recent mine plan. Composite samples representing major lithologies and a range of head grades were prepared (0.60 to 2.0 g/t Au and 0.5 to 20 g/t Ag). The minimum and maximum grades aligned with expected plant feed for the first nine years of production.

Bulk mineralogy on select composites showed the main sulphide mineral was pyrite, ranging from 5.3 to 7.7%, with traces of chalcopyrite, sphalerite, and galena. Gold deportment studies indicated 5 to 12% of the gold is sub-microscopic; 8 to 14% of the gold is locked in <11 µm size fractions; 42 to 64% of the gold is exposed and 22 to 32% is liberated. A host of telluride minerals exist in the microscopic size range, with petzite the most dominant. Gold and electrum occur in minor amounts.

Comminution testing showed that the materials tested are considered very soft to medium in competency, with SMC test A*b values ranging from 40 to 124 and SPI test results from 7 to 67 min. Conventional Bond tests showed significant variation in hardness, with Bond rod mill work indices ranging 9 to 15 kWh/t and Bond ball mill work indices ranging from 8 to 18 kWh/t, at a closing screen size of 150 µm.

Two parallel flowsheets were evaluated, following the results from the previous studies: flotation + concentrate and tailings leaching versus whole ore leaching. The recommended flowsheet for this study is flotation with concentrate/tailings leaching.

Whole ore cyanide leach tests showed relatively poor extraction at a grind size of 80% passing 75 µm or greater using aggressive leach conditions to combat the effects of the telluride minerals. Gold leach extractions ranged from 52 to 72%. At a finer grind of 80% passing 60 µm, gold extractions ranged from 64 to 84%.

Rougher flotation tests showed high sulphide recovery was generally achieved within eight minutes. Excessive foaming was observed in some samples. This was considered attributable to a drilling compound added to the core, to aid core recovery (this was also commented on in the 2019 updated PEA report for the Project, which tested samples from the same drilling program). High mass pull was observed in these samples. A cleaning stage reduced the mass pull reporting to concentrate regrind. Flotation recoveries to cleaner concentrate ranged from 55 to 83% for gold, 55 to 90% for silver and 75 to 98% for sulphur at a target mass pull of 15% or less. Leaching of flotation tails is required to attain acceptable gold recovery. Tailings samples showed very high leach extractions in general.

Flotation concentrate gold extraction showed significant benefit from finer regrinding to an 80% passing size of 15 to 17 µm. Particularly high concentrate leach residue grades were observed at 80% passing 25 µm. Flotation concentrate gold extractions ranged from 62 to 97%, somewhat dependent on gold head grade. Flotation tails gold extractions ranged from 52 to 94%.

Overall plant gold recoveries are predicted to average 86% for head grades of 0.8 to 1.22 g/t Au. Overall plant recoveries for silver are predicted to range from 85 to 92% for head grades of 3.2 to 8.3 g/t Ag.

Cyanide detoxification tests achieved <1 mg/L CN_{WAD}, with favourable reagent consumption rates.

Mercury grades were in the range of <0.3 to 8 g/t in the flotation feed. A retort with gas collection system was incorporated into the plant design to manage and control mercury in the process. Arsenic is present in the feed at concentrations up to 30 g/t and is not expected to be problematic in processing. No other elements were noted that may cause issues in the process plant or concerns with product marketability.

Thickening and filtration of cyanide detoxified slurry showed a moisture content of 18.5% (by weight) was achieved with high-rate thickening followed by pressing and drying using a conventional plate and frame filter press. A moisture content of 15% was achieved when employing a membrane squeeze in addition to pressing and drying in a plate and frame filter.

13.6 Recommendations

1. Future drilling should be done using drill mud additives that have been demonstrated to have minimal impact on metallurgical testwork. A bulk sample might be considered to avoid the issue of drilling compound modifying reagents.
2. Investigate the impact of drilling mud additives on flotation mass pull with the objective of reducing flotation circuit size and regrind power requirements.
3. Further optimize concentrate leach reagents and consider reductions in leach extraction time. This includes reducing the number of concentrate leach adsorption tanks and recover residual gold/silver in solution using the flotation tails CIP circuit.

4. Optimize combined tails residual cyanide levels and aim to reduce cyanide detoxification retention time.
5. Conduct a full Feasibility Study metallurgical testwork program incorporating variability and production composite testwork. This includes dewatering/filtering tests on the final tailings material.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The Mineral Resource statement presented herein represents the fourth mineral resource evaluation that has been prepared for the Springpole Gold Project in accordance with NI 43-101 and the current CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014).

There is a total of 662 drill holes in the Springpole database provided to SRK. The current mineral resource model prepared by SRK utilizes results from 404 core boreholes drilled by previous owners of the property during the period of 2003 to 2013, and seven holes drilled by First Mining in 2016 and 2020. Geotechnical drill holes that were completed in 2018 and 2020 to test the integrity of the lakebed for cofferdams and test areas of proposed mining infrastructure were not included in the mineral resource estimate presented in this report as they are outside the resource area. The resource estimation work was completed by Dr. Gilles Arseneau, P.Geo (APEGBC #23474), an independent Qualified Person as this term is defined in NI 43-101. The effective date of the resource statement is June 26, 2020.

This section describes the resource estimation methodology and summarizes the key assumptions considered by SRK. In the opinion of SRK, the resource evaluation reported herein is a reasonable representation of the global gold and silver resources found in the Springpole Gold Project at the current level of sampling. The mineral resources were estimated in conformity with the CIM Estimation of Mineral Resource and Mineral Reserves Best Practices guidelines and are reported in accordance with the 2014 CIM Definition Standards.

The database used to estimate the Mineral Resources was audited by SRK. SRK is of the opinion that the current drilling information is sufficiently reliable to interpret with confidence the boundaries for porphyry gold mineralization and that the assay dataset is sufficiently reliable to support mineral resource estimation.

GEMS (6.7) was used to construct the geological solids, prepare assay data for geostatistical analysis, construct the block model, estimate metal grades, and tabulate mineral resources. The geostatistical software SAGE2001 was used for variography.

14.2 Resource Estimation Procedures

The resource evaluation methodology involved the following procedures:

- database compilation and verification
- construction of wireframe models for the boundaries of the Springpole Gold mineralization
- definition of resource domains
- data compositing and capping for geostatistical analysis and variography

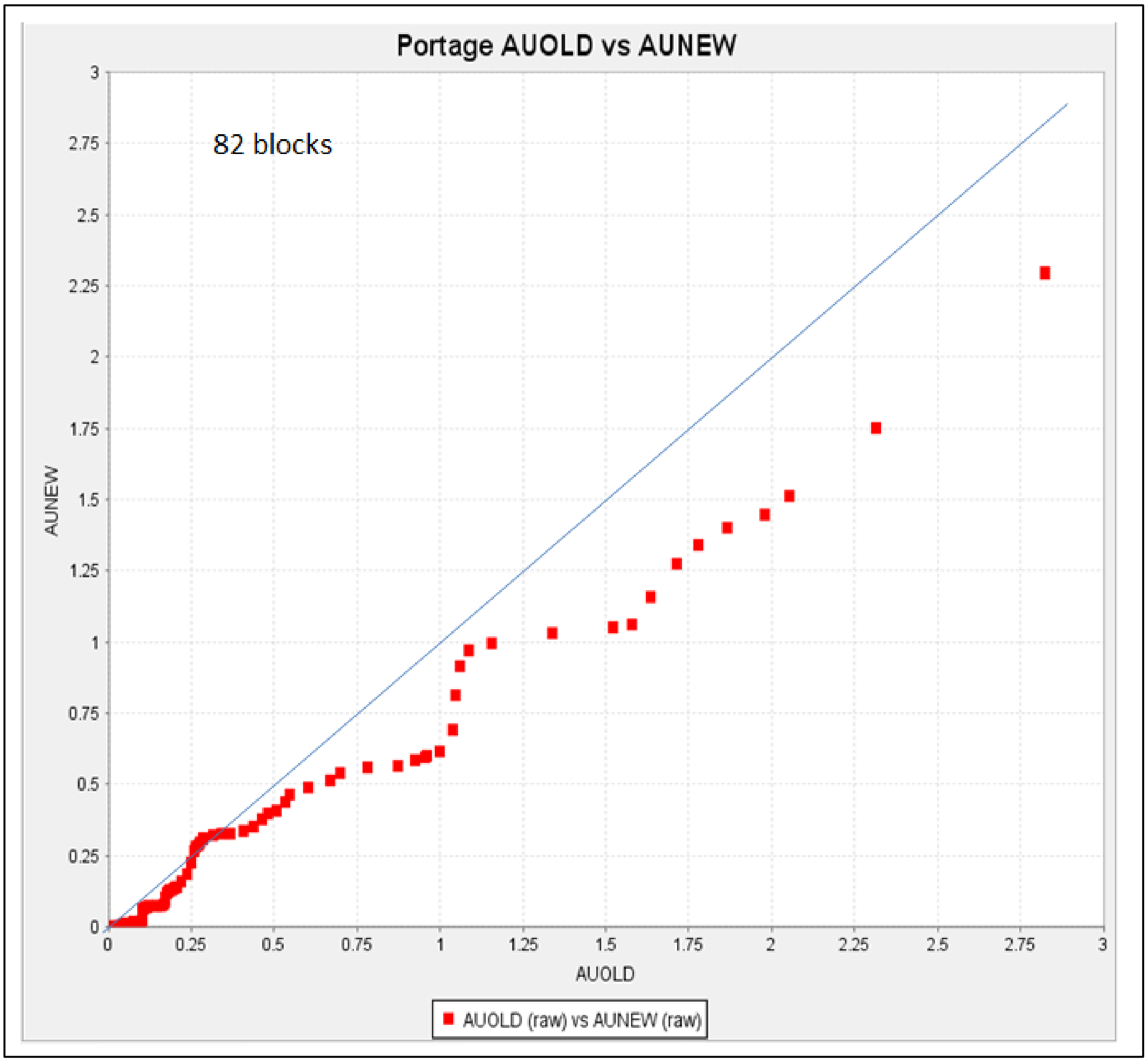
- block modelling and grade interpolation
- resource classification and validation
- assessment of "reasonable prospects for economic extraction" and selection of appropriate cut-off grades (COGs)
- preparation of the mineral resource statement

14.3 Drill Hole Database

The Springpole Gold Project currently consists of three separate mineralized zones: East Extension, Camp or Main, and Portage. The Portage zone is by far the largest of the three and represents more than 90% of the stated resource.

The entire Springpole database provided to SRK consists of 662 drill holes totalling 182,432 m. Of these, 258 drill holes were not used in the estimation, 186 holes did not intersect the mineralized zones, two metallurgical holes had no assay data due to having been completely sampled for metallurgical testing and 70 historical BL series holes were discarded because of apparent bias due to poor core recovery (Figure 14-1).

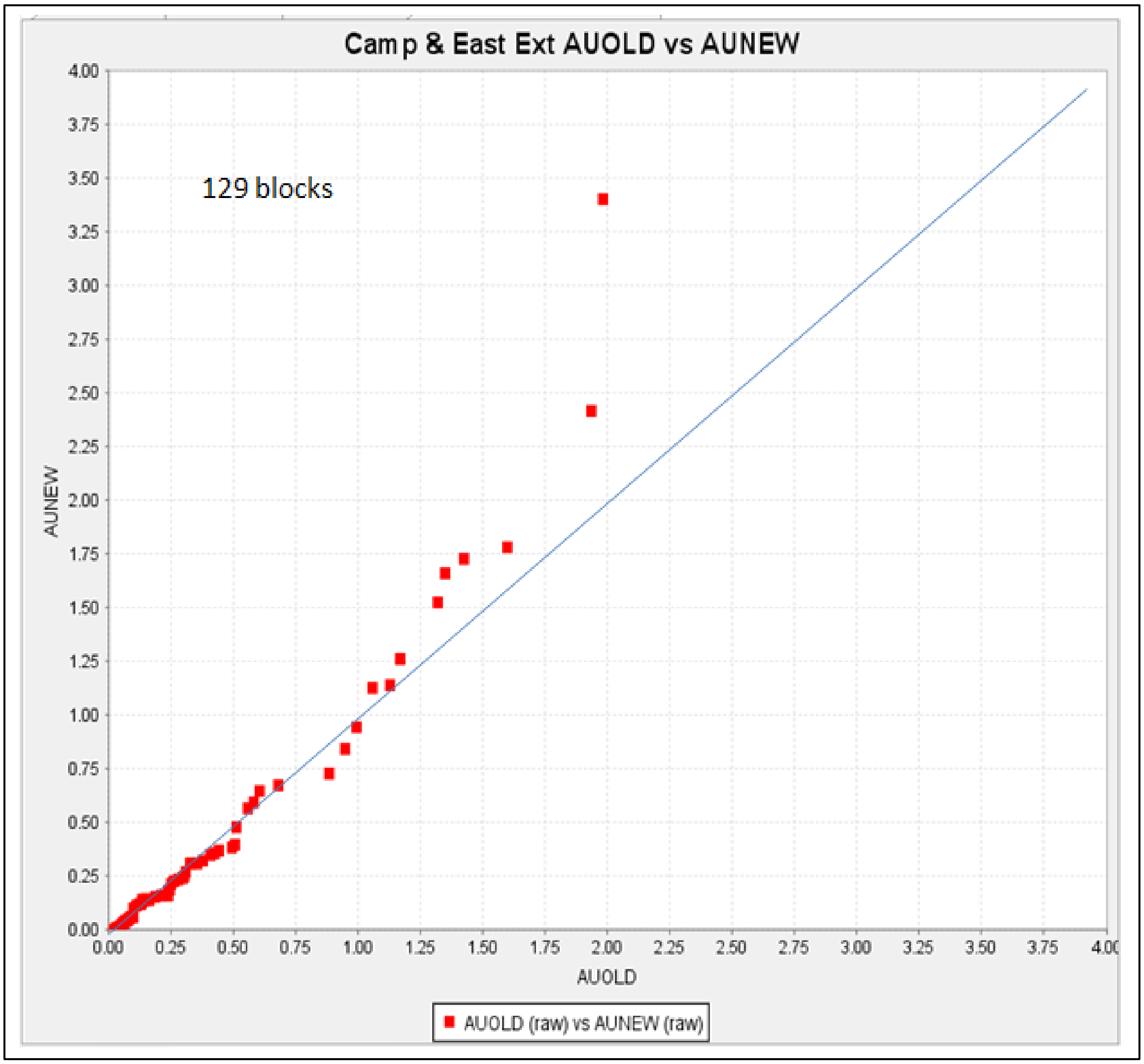
Figure 14-1: Comparison of Historical and Recent Drilling for the Portage Zone



Source: SRK, 2019

Consequently, because of the good agreement between the recent and old drilling for the Camp and East Extension zones (Figure 14-2), it was decided that all historical drilling from 1986 to present would be included for estimation of these two zones. The Portage zone was estimated using only 2003 and later drill holes.

Figure 14-2: Comparison of Historical and Recent Drilling for the Camp and East Extension Zones



Source: SRK, 2019

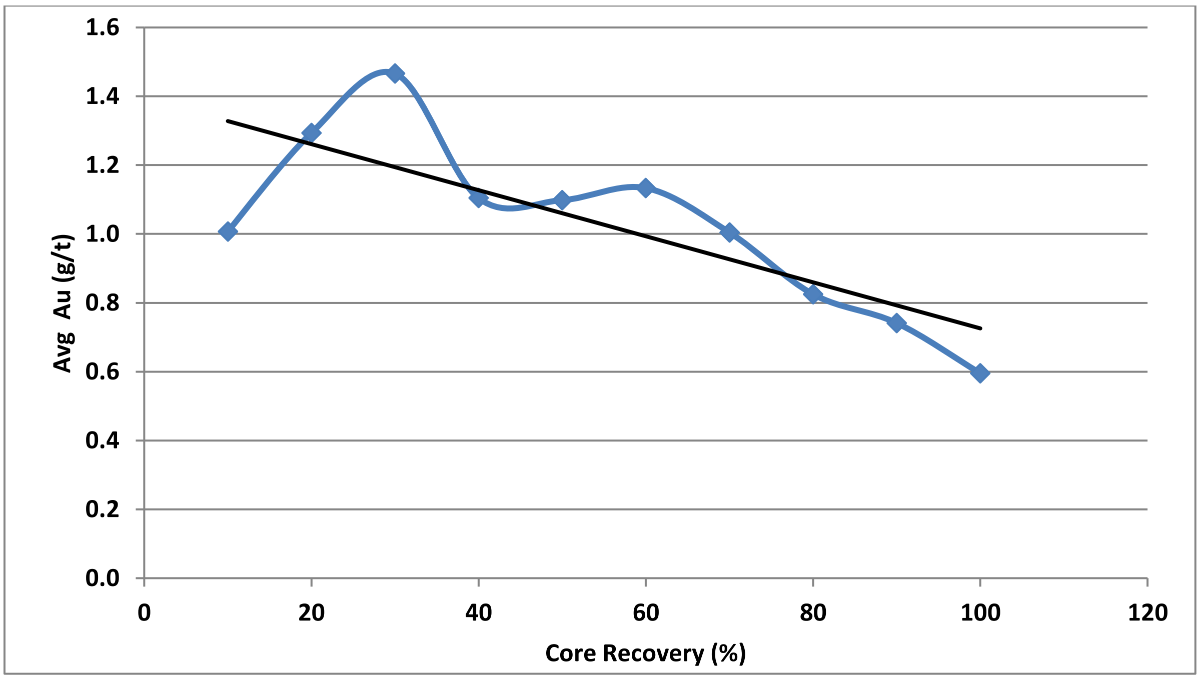
14.4 Core Recovery

Drill core recovery for both the East Extension and Camp zones was generally very good with average recovery recorded as approximately 97%. For Portage, with areas of intense argillic and potassic alteration, core recovery was a much more significant issue, primarily affecting near surface intervals and intervals that appear to intersect a narrow zone of intense biotitic alteration.

SRK conducted studies to determine if there was any significant bias indicated, either high or low, as a function of core recovery. To a certain extent it was anticipated that more intense zones of alteration could also often reflect more intense mineralization.

Core recovery was generally recorded in 3 m intervals, with some data recorded in 1.5 m intervals. Consequently, for this analysis, it was decided to composite the core recovery values to the 3 m sample lengths and compare them with assay grades. The comparison indicates that the gold grade is generally lower with the increased recoveries (Figure 14-3). For this reason, SRK decided to separately model areas of low core recoveries and treat those volumes as having hard boundaries during grade interpolation.

Figure 14-3: Gold Grade versus Core Recovery Relationship



Source: SRK, 2019

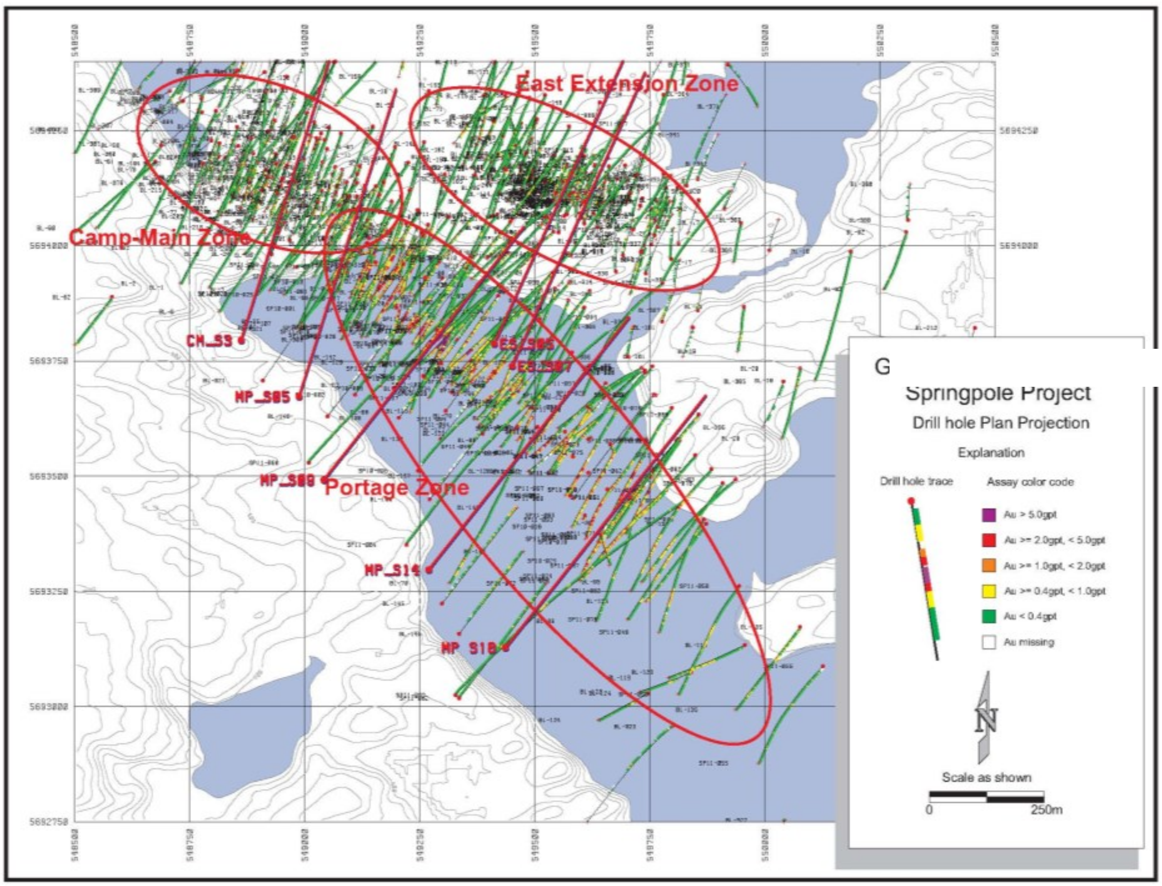
14.5 Geological Domains

The Camp zone lies to the north and, where the two domains overlap, above the Portage zone. The Camp zone strikes approximately 120° (N120°E) and part of the zone is very similar in character to the East Extension with highly erratic grades showing very little spatial organization.

The Portage zone is by far the most significant domain, extending from beneath the southern extent of the Camp zone for more than 1,500 m to the southeast. Other than location, the Portage zone exhibits few similarities with the other two domains. Also, in contrast with East Extension and Camp, Portage has significant silver mineralization closely associated with gold. Drill-tested mineralization is extremely continuous with very little evidence of isolated erratic higher-grade intervals. As drilled, Portage represents a zone of largely disseminated mineralization striking 135° (S45°E) and extending from the surface to a depth of over 400 m, on average approximately 150 m in width and over 1,500 m in length.

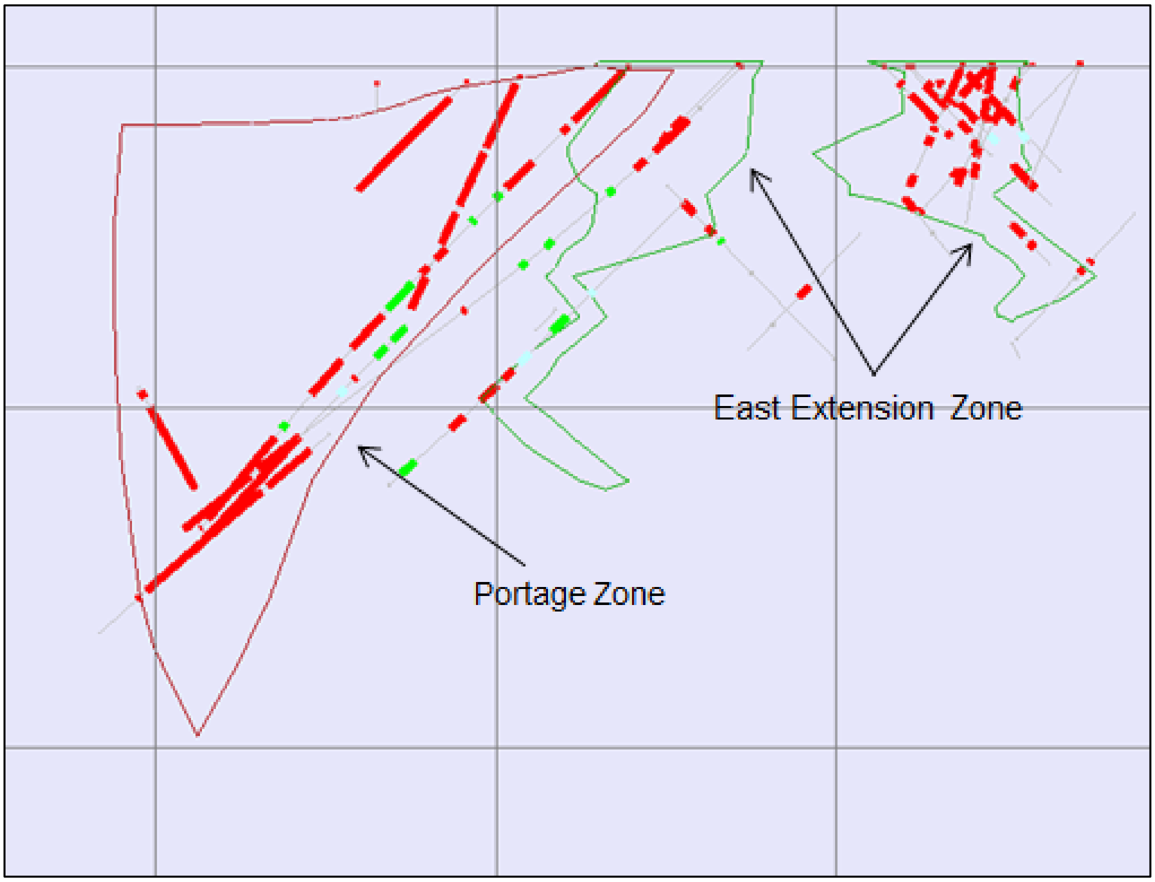
Geological domains were defined on sections spaced at 50 m intervals and a cut-off of 0.2 g/t was used to identify the geological domains on sections. Figure 14-4 shows the Springpole drill plan with the three geological domains and Figure 14-5 shows the domain boundaries on a typical section.

Figure 14-4: Geological Domains for Springpole Gold Project



Modified from: Gold Canyon, 2011

Figure 14-5: Cross Section 1100NW Looking NW Showing Portage and East Extension Domains



Source: SRK, 2019
Note: Grid is 200 by 220 m. Green drill hole traces are > 0.2 g/t and red traces are > 0.3 g/t Au.

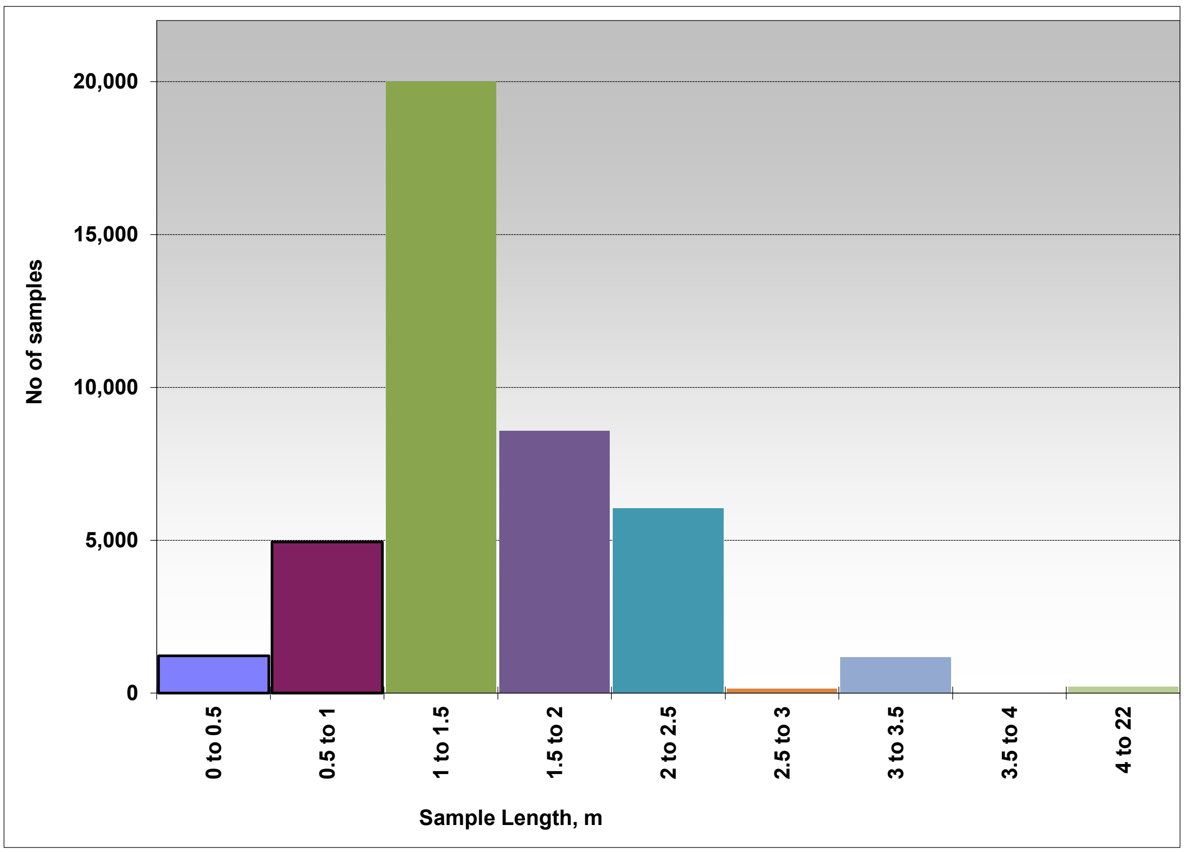
14.6 Surface Topography

Topography was provided in the form of a Drawing Interchange Format file containing data from a 2011 light detection and ranging (LiDAR) survey with vertical precision of 1 m. The topographic surface beneath the small portion of the lake overlying the Portage zone was established by ground penetrating radar, echo sounder and sub-bottom profiling surveys conducted by Terrasond Ltd. of Palmer, Alaska, from the frozen surface (March 2011) and water lake surface (June 2011). These multiple surfaces were then merged to create a continuous surface to constrain the top of the block model. Overburden surface was modelled by extracting the base of the overburden from all available drill hole logs and generating a surface by simple triangulation of drill hole points.

14.7 Compositing

An analysis of the sample lengths within the mineralized domains shows that sample lengths are variable ranging from a low of 0.1 m to a maximum of 21 m; however, the majority of the samples are between 0.5 and 3 m in length with the largest proportion of the samples at 1 m in length (Figure 14-6). Most samples, 99%, are less than 3 m in length and for this reason SRK decided to composite all assays to a 3 m length within the mineralized envelopes. Compositing was generated from the drill collars and compositing was interrupted at domain boundaries. The compositing process generated 24,372 composites. A total of 274 composites with length less than 1.5 m were discarded from the database prior to resource estimation.

Figure 14-6: Histogram of Sample Lengths within Mineralized Domains



Source: SRK, 2019

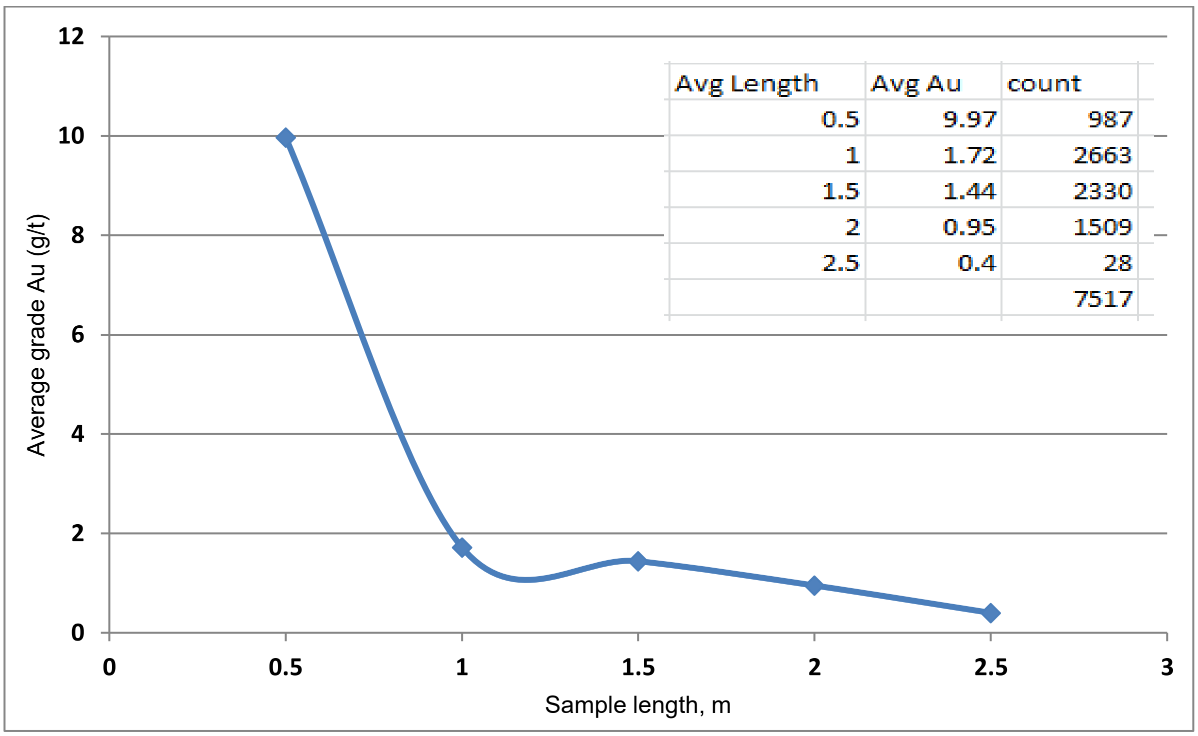
14.8 Grade Capping

The primary goal of grade capping is to identify and restrict the influence of suspected “outlier” grades in an estimate.

Grade capping for the Springpole Gold Project was carried out in two stages. First the assay dataset was investigated to determine if sample length could bias the average grade. An analysis of gold grade against sample length seems to indicate that sample length of less than 1 m has a significantly higher average grade than other sample lengths, indicating these samples were taken over a specific geological domain, perhaps quartz veins or narrow siliceous zones with visible gold (Figure 14-7). Most short sample lengths seem to have been taken from the Camp and East Extension zones; for this reason, SRK decided to treat these short sample lengths as a separate statistical population and capped these short assays prior to compositing. SRK capped all gold assays for sample lengths less than 1 m to 100 g/t Au prior to compositing.

All assays were then composited to 3 m lengths and all 3 m composites were evaluated for outliers by examining their distribution on cumulative probability plots and capped as outlined in Table 14-1.

Figure 14-7: Comparison of Sample Length and Average Gold Grade



Avg Length	Avg Au	count
0.5	9.97	987
1	1.72	2663
1.5	1.44	2330
2	0.95	1509
2.5	0.4	28
		7517

Source: SRK, 2019

Table 14-1: Capping Levels for Springpole

Element	3 m Composite Capping Level
Au	25 g/t
Ag	200 g/t

14.9 Statistical Analysis and Variography

Statistical analyses were carried out on both the raw assay data and on the 3 m composited data. There is a total of 122,089 entries in the drill hole assay table for the Springpole Gold Project. Of these, 52,581 are within the interpreted wireframes representing the three mineralized domains. Some 8,191 historical assays within the mineralized domains were rejected because of uncertainties relating to quality control procedures. Of the data accepted, 288 samples were missing gold and silver assays because of missing core resulting from poor recovery. Data for these cores were omitted from the statistical analysis presented in Table 14-2. Statistical data for the 3 m composited data are presented in Table 14-3 excluding the 288 missing samples due to poor core recovery.

Table 14-2: Basic Univariate Statistical Information for Raw Assay Data

Zone	Max (g/t)	Min	Mean (g/t)	Std. Dev.	CoV	Count
East Extension Gold	1568	0	1.65	27.23	16.5	8,195
Camp Gold	459	0	1.16	10.6	9.16	3,838
Portage Gold	207	0	0.78	2.41	3.08	40,260
Portage Gold	300	0	4.03	10.79	2.66	31,554

Note: 242 samples in the Portage Zone do not have gold assays and 8,948 samples have no silver assays, the missing data were excluded from the above table.

Table 14-3: Basic Univariate Statistical Information for 3 m Composites

Zone	Max (g/t)	Missing Values	Mean (g/t)	Std. Dev.	CoV	Count[1]
East Extension Gold	269.27	153	0.88	6.03	6.88	3312
East Extension Capped Gold	25	153	0.7	2.43	3.49	3312
Camp Gold	89.65	85	0.81	3.36	4.16	1447
Camp Capped Gold	25	85	0.75	2.31	3.08	1447
Portage Gold	95.3	695	0.77	1.84	2.38	18,680
Portage Capped Gold	25	695	0.76	1.34	1.77	18,680
Portage Gold	273	4282	4.29	9.47	2.20	14,888
Portage Capped Gold	200	4282	4.28	9.24	2.16	14,888

Note: In addition, there are 933 composites with missing gold assays and 4,282 composites with missing silver assays, these were excluded from the table above and were not used during grade interpolation.

Spatial continuity of gold and silver was evaluated with correlograms developed using SAGE 2001 version 1.08. The correlogram measures the correlation between data values as a function of their separation distance and direction. The distance at which the correlogram is close to zero is called the "range of correlation" or simply the range. The range of the correlogram corresponds roughly to the more qualitative notion of the "range of influence" of a sample or composite.

Variographic analysis was completed for gold in the Portage, Camp, and East Extension zones and for silver in the Portage zone. Directional correlograms were generated for composited data at 30° increments along horizontal azimuths. For each azimuth, correlograms were calculated at dips of 0°, 30°, and 60°.

A vertical correlogram was also calculated. Using information from these 37 correlograms, SAGE determines the best fit model using the least square fit method. The correlogram model is described by the nugget (C_0) and two nested structure variance contributions (C_1, C_2) with ranges of the variance contributions and the model type (spherical or exponential). After fitting the variance parameters, the algorithm then fits an ellipsoid to the 37 ranges from the directional models for each structure. The final models of anisotropy are given by the lengths and orientations of the axes of the ellipsoids.

The correlogram models applied in the resource estimates in each domain are presented in Table 14-4.

Table 14-4: Gold and Silver Spherical Correlogram Parameters by Domain

Domain	Metal	Nugget	Sill	Gemcom Rotations (RRR rule)			Ranges a1, a2		
		C_0	C_1, C_2	around Z	around Y	around Z	X-Rot	Y-Rot	Z-Rot
Camp	Au	0.3	0.67	-27	57	52	26	8	5
			0.03	-27	57	52	61	57	180
East Extension	Au	0.3	0.48	-6	-67	-72	7	11	15
			0.22	-6	-67	-72	20	49	150
Portage	Au	0.19	0.56	31	8	34	20	40	20
			0.25	31	8	34	60	138	168
Portage	Ag	0.1	0.61	-48	30	27	22	9	18
			0.29	-48	30	27	100	76	174

14.10 Block Model and Grade Estimation

Block modelling was carried out in GEMS (6.4) software by Dr. Gilles Arseneau, P.Geo., an Associate Consultant with SRK. Block estimates were carried out in 10 by 10 by 6 m blocks using a percent model to weight partial blocks situated at zone boundaries. Block model parameters are defined in Table 14-5.

Table 14-5: Block Model Setup Parameters

	Model Origin (UTM - WGS 84)	Block Size (m)	No. of blocks
Easting	548,500	10	220
Northing	5,692,400	10	210
Elevation	418	6	90

14.10.1 Gold and Silver Grade Models

Grades were estimated by ordinary kriging with a minimum of 4 and a maximum of 15 composites with no more than three composites permitted from a single drill hole. Grade interpolations were carried out in three passes with each successive pass using a larger search radius than the preceding pass and only estimating the blocks that had not been interpolated by the previous pass. Table 14-6 summarizes the search parameters for each interpolation pass.

Table 14-6: Search Parameters by Zone and Metal

Metal	Zone	Pass	Rotation			Search Ellipse Size			Number of Composites		Max. Samples per DDH
			Z	Y	Z	X (m)	Y (m)	Z (m)	Min.	Max.	
Au	Camp	1	-84	7	-32	20	30	20	4	15	3
Au	Camp	2	-84	7	-32	40	60	60	4	15	3
Au	Camp	3	-84	7	-32	60	138	168	4	15	3
Au	East Ext	1	-84	7	-32	20	30	20	4	15	3
Au	East Ext	2	-84	7	-32	40	60	60	4	15	3
Au	East Ext	3	-84	7	-32	60	138	168	4	15	3
Au	Portage	1	-84	7	-32	20	30	20	4	15	3
Au	Portage	2	-84	7	-32	40	60	60	4	15	3
Au	Portage	3	-84	7	-32	60	138	168	4	15	3
Ag	Portage	1	-48	30	27	20	30	20	4	15	3
Ag	Portage	2	-48	30	27	40	60	60	4	15	3
Ag	Portage	3	-48	30	27	100	76	100	4	15	3

Uncapped gold was also estimated for all three domains for comparison against the capped results. The capped estimates were used for use in resource reporting and classification.

14.10.2 Bulk Density Model

There are 611 bulk density measurements in the Springpole database with an average of 2.61 t/m^3. SRK is of the opinion that these are sufficient to estimate a Mineral Resource.

SRK decided to estimate the bulk density by inverse distance squared where dataset was nearby or assign an average density to un-estimated blocks, as presented in Table 14-7.

Table 14-7: Bulk Density of Un-estimated Blocks in the Model

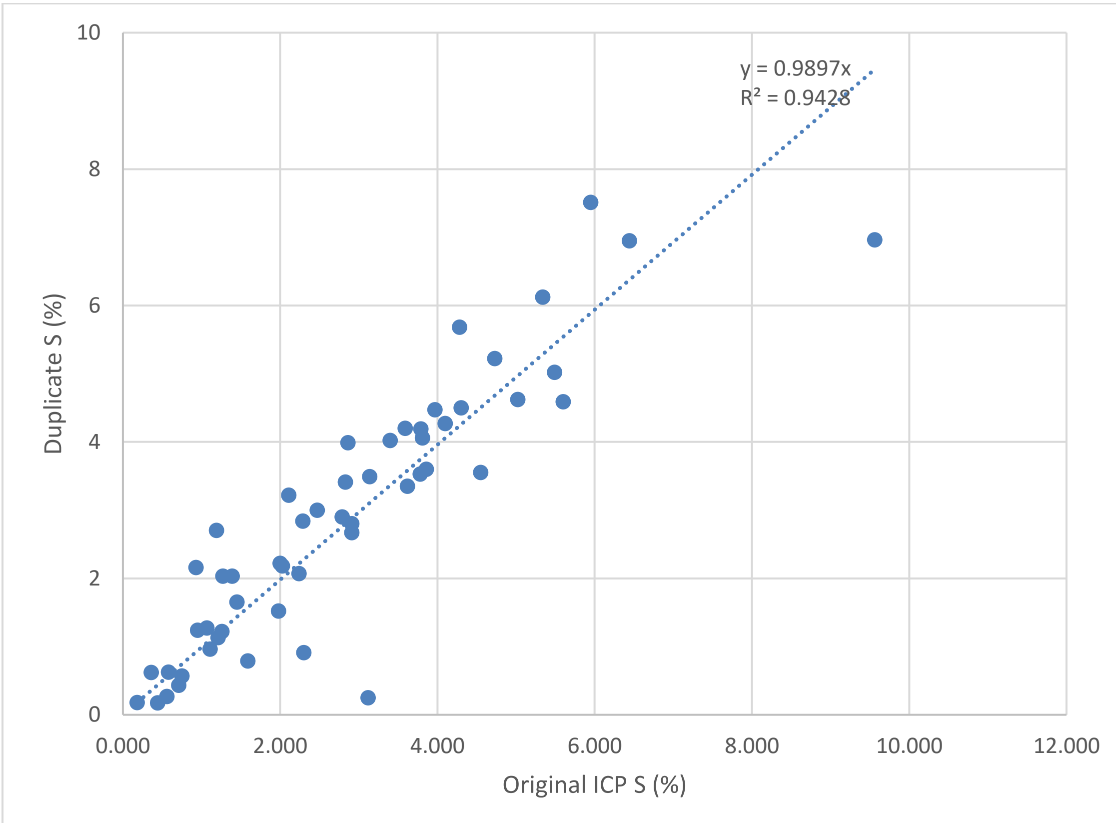
Zone	Average Density of Un-Estimated Blocks (t/m^3)
Camp	2.73
East Extension	2.91
Portage	2.58
Waste rock	2.74
Overburden	1.9

14.10.3 Total Sulphur Model

In addition to gold and silver grades, total sulphur was estimated in the block model to facilitate the determination of acid base accounting (ABA) of the mineralized and waste rock.

Sulphur was determined from 14,731 samples, 5,862 of these were from the historical ICP assay database, 511 were from the acid rock drainage (ARD) test work program, and 8,358 were from samples of existing core collected in the winter of 2019-20 program. Samples collected during the 2019-20 winter program were analyzed by SGS using a LECO sulphur analyser with high temperature combustion. From the samples collected in 2019-20, 53 intervals have corresponding historical ICP assay values. SRK examined the duplicate samples and concluded that while the ICP assays seemed to be slightly lower than the total sulphur data that the bias was not significant enough to reject the historical data (Figure 14-8). SRK also noted that the historical ICP sulphur data had an upper detection limit set at 5% sulphur and that 193 samples collected during the 2010 drilling program were not re-assayed to determine their correct sulphur content, effectively capping these samples at 5% sulphur.

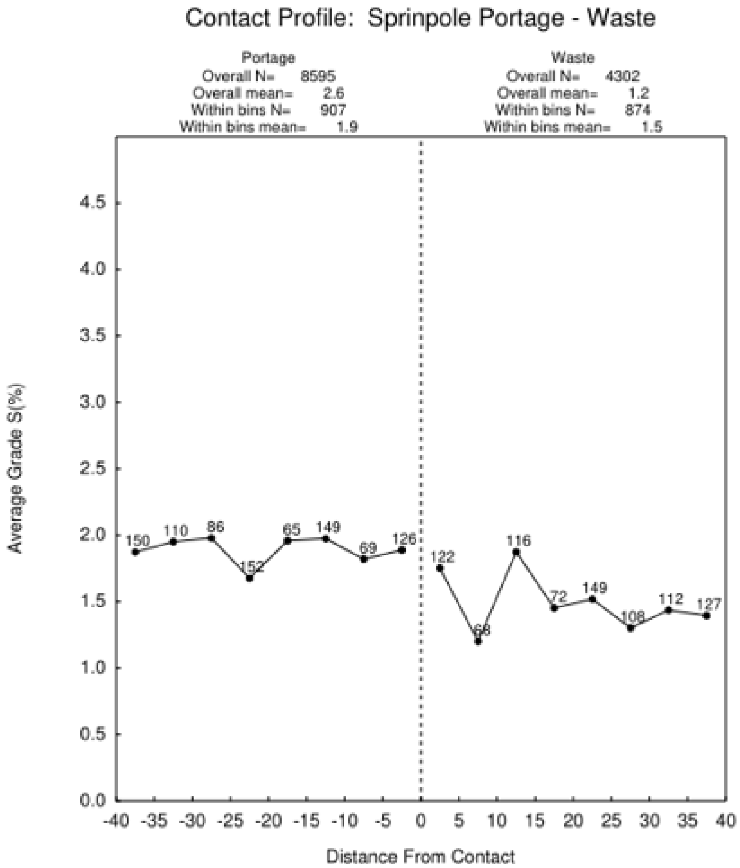
Figure 14-8: Comparison of Historical S% by ICP with Re-assayed Total S%



Source: SRK, 2021

SRK decided to evaluate the sulphur data by mineralized zones and rock type to determine if sulphur content correlated with mineralization or rock type. The analysis showed that sulphur content is very similar inside and outside of the Portage zone and that the mineralized zone contact is not a hard boundary with respect to sulphur content (Figure 14-9).

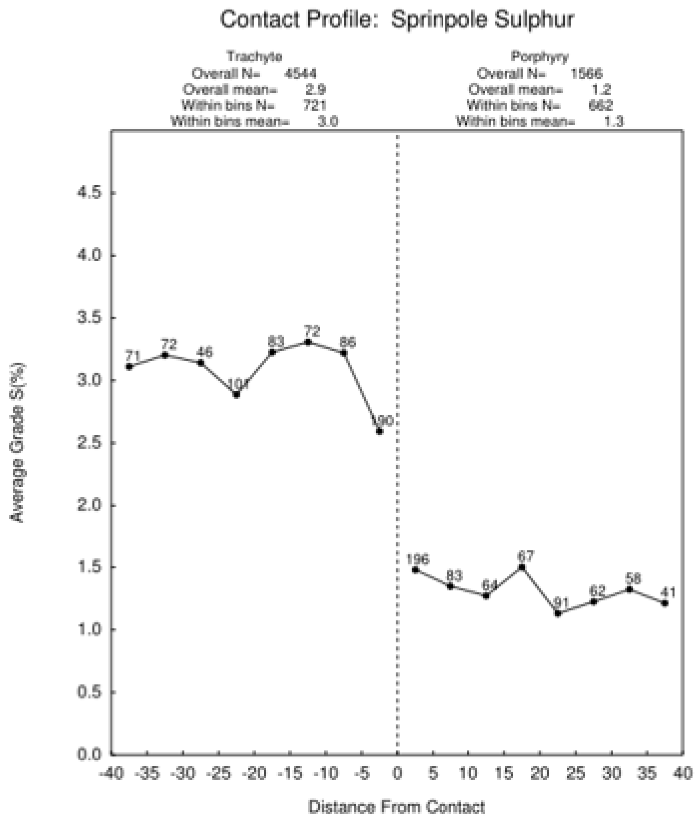
Figure 14-9: Contact Profile for Sulphur for the Portage Mineralized Zone



Source: SRK, 2021

However, sulphur seems to correlate well with rock type, with some rock types having higher sulphur contents than surrounding rock units (Figure 14-10). For this reason, SRK treated some rock type contacts as having a hard boundary and some as soft boundaries depending on their contact profiles (Table 14-8).

Figure 14-10:Sulphur Contact Profile between Trachyte and Porphyry



Source: SRK, 2021

Table 14-8: Boundary Type used for Total Sulphur Determination

	Breccia	Meta Sediment	Porphyry	Trachyte
Breccia		Hard	Hard	Soft
Meta Sediment	Hard		Soft	Soft
Porphyry	Hard	Soft		Hard
Trachyte	Soft	Soft	Hard	

SRK estimated total sulphur in the model by ordinary kriging in three separate passes with increasing search radius (Table 14-9).

Table 14-9: Sulphur Interpolation Parameters

	Pass Range (m)			Number of Samples	
	1	2	3	Minimum	Maximum
Easting	60	120	178	4	15
Northing	45	90	139	4	15
Elevation	90	180	280	4	15

14.10.4 Acid Base Accounting Model

In addition to the Total Sulphur model, SRK estimated parameters to facilitate the determination of acid generation potential of the rock within the open pit. A total of 618 samples were collected from drill core and analyzed to determine their acid generation potential in order to quantify if the rock was PAG or NAG.

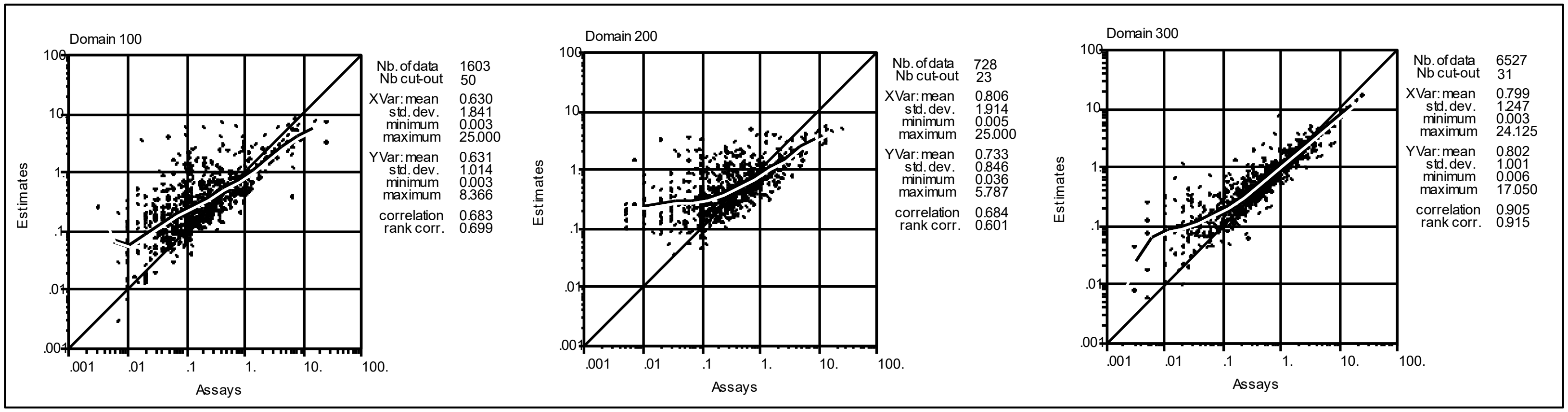
Because of the sparsity of the ABA data, SRK decided to estimate the values using a nearest neighbour interpolation method. All blocks were then coded as PAG if the sulphide sulphur net potential ratio (SNPR) were less or equal to 2 and coded as NAG if the SNPR was greater than 2.

14.11 Model Validation

The Springpole resource block model was validated by completing a series of visual inspections. It was additionally validated by comparing local “well-informed” block grades with composites contained within those blocks, and by comparing average assay grades with average block estimates along different directions – swath plots.

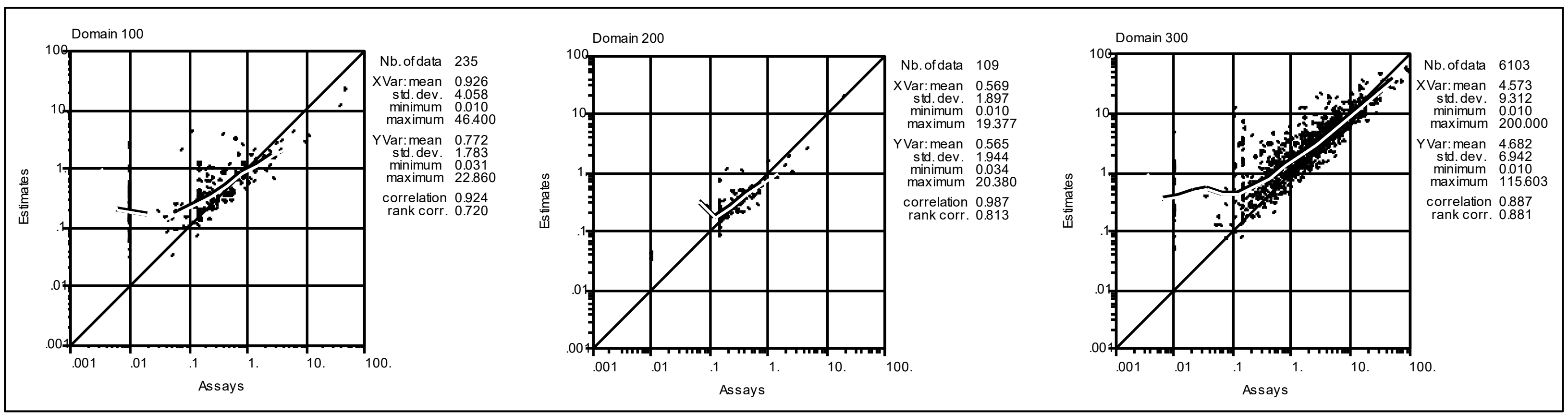
Figure 14-11 shows a comparison of estimated gold block grades with borehole composite assay data contained within those blocks within the mineralized domains. Figure 14-12 illustrates the same comparison for the silver grades. On average, the estimated blocks are similar to the composite data, although there is a large scatter of points around the x = y line. This scatter is typical of smoothed block estimates compared to the more variable assay data used to estimate those blocks. The thick white line that runs through the middle of the cloud is the result of a piece-wise linear regression smoother.

Figure 14-11: Comparison of Gold Grades for Well-Informed Blocks



Source: SRK, 2019

Figure 14-12: Comparison of Silver Grades for Well-Informed Blocks



Source: SRK, 2019

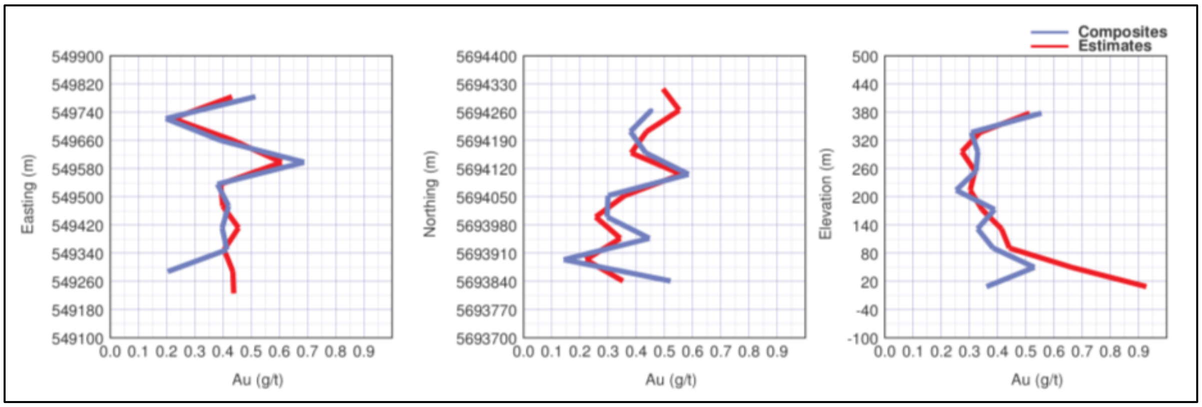
Note that there are relatively few data for silver for the East Extension (domain 100) and Camp zone (domain 200). This is due to the fact that only the Gold Canyon drill holes had silver assay data for these two mineralized zones.

As a final check, average composite grades and average block estimates were compared along different directions. This involved calculating de-clustered average composite grades and comparing them with average block estimates along east-west, north-south, and horizontal swaths.

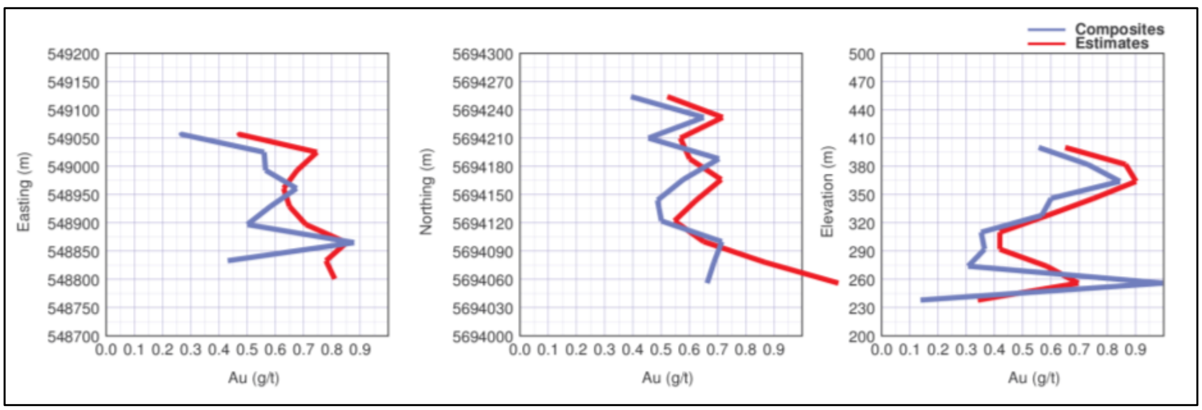
Figure 14-13 shows the swath plots in the three mineralized zones, and Figure 14-14 shows the swath plot for silver within the Portage zone. The average composite grades and the average estimated block grades are quite similar in all directions. Similar behaviour was documented for all other mineralized zones. Overall, the validation shows that the current resource estimate is a good reflection of drill hole composited data for the Springpole Gold Project.

Figure 14-13: Swath Plots for Gold re (a) the East Extension, (b) the Camp, and (c) the Portage Zone

(a)



(b)



(c)

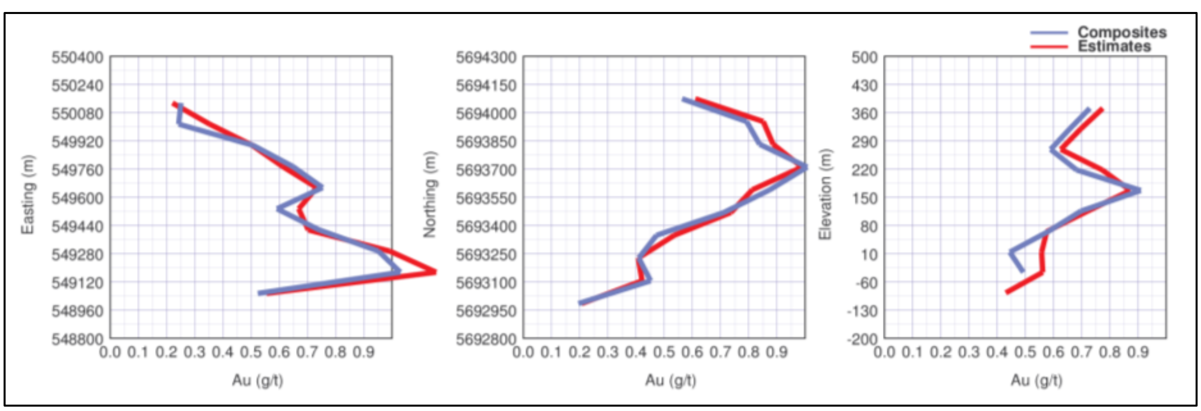
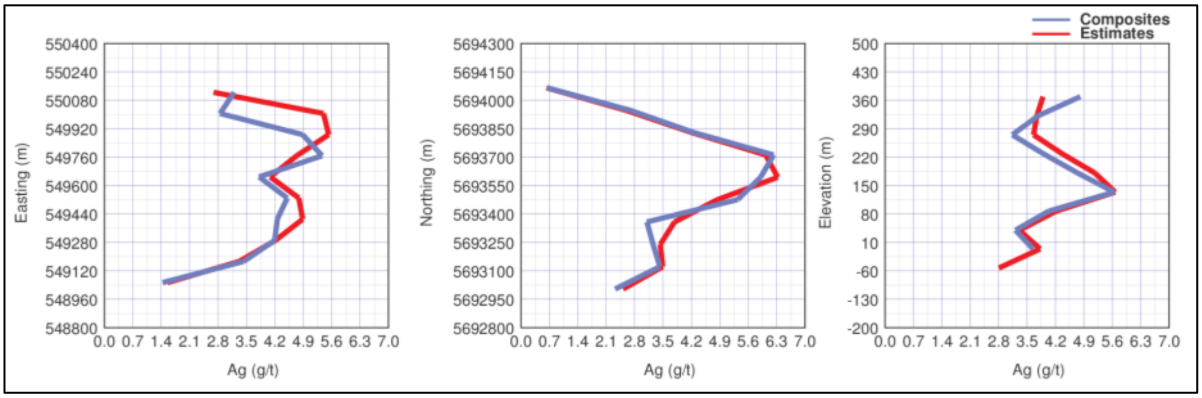


Figure 14-14: Swath Plot for Silver within the Portage Zone



Source: SRK, 2019

14.12 Mineral Resource Classification

Block model quantities and grade estimates for the Springpole Gold Project were classified by Dr. Gilles Arseneau, Ph.D., P.Geo. (APEGBC #23474), an independent Qualified Person for the purposes of NI 43-101. The classification was completed according to the current CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014).

Mineral resource classification is typically a subjective concept; industry best practices suggest that mineral resource classification should consider the confidence in the geological continuity of the mineralized structures, the quality and quantity of exploration data supporting the estimates, and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating these concepts to delineate regular areas at similar resource classification.

SRK is satisfied the geological modelling honours the current geological information and knowledge. The location of the samples and the assay dataset is sufficiently reliable to support resource evaluation. The sampling information was acquired primarily by core drilling on sections spaced at 50 m.

The Mineral Resources were classified according to the following rules:

1. For the East Extension zone, any blocks estimated during Pass 1 or Pass 2 with at least two drill holes and six composites were classified as Indicated Mineral Resources. All other estimated blocks were classified as Inferred Mineral Resources.
2. The Portage and Camp classification was based solely on the gold estimate. Silver, as a minor by-product, carries the classification associated with the gold. Any blocks that were estimated during Pass 1 or Pass 2 with at least two drill holes and six composites were classified as Indicated Mineral Resources. All other interpolated blocks were classified as Inferred Mineral Resources.

14.13 Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) defines a mineral resource as:

"A Mineral Resource is 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. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling".

The "material of economic interest" refers to diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals.

The "reasonable prospects for economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade taking into account extraction scenarios and processing recoveries. In order to meet this requirement, SRK considers that major portions of the Springpole Gold Project are amenable for open pit extraction.

To determine the quantities of material offering "reasonable prospects for economic extraction" by an open pit, SRK used a pit optimizer and reasonable mining assumptions to evaluate the proportions of the block model (Indicated and Inferred blocks) that could be "reasonably expected" to be mined from an open pit.

The optimization parameters were selected based on experience and benchmarking against similar projects (Table 14-10). The reader is cautioned that the results from the pit optimization are used solely for the purpose of testing the "reasonable prospects for economic extraction" by an open pit and do not represent an attempt to estimate mineral reserves. The results are used as a guide to assist in the preparation of a Mineral Resource statement and to select an appropriate resource reporting cut-off grade.

Table 14-10: Assumptions Considered for Conceptual Open Pit Resource Optimization

Parameter	Units	Value
Au Price	$/oz	1550
Ag Price	$/oz	20
Exchange Rate	$US/$CDN	0.77
Mining Cost	$/t mined	1.62 (ore)
Processing	$/t of feed	15.38
General and Administrative	$/t of feed	1.00
Overall Pit Slope	degrees	35 to 50 based on domains
Au Process Recovery	percent	88
Ag Process Recovery	percent	93
In Situ COG	g/t	0.30 Au

SRK considers that the blocks located within the conceptual pit envelope show "reasonable prospects for economic extraction" and can be reported as a Mineral Resource (Table 14-11).

Table 14-11: Mineral Resource Statement* Inclusive of Mineral Reserves (effective July 30, 2020)

Category	Quantity	Grade		Metal	
	(Mt)	Au (g/t)	Ag (g/t)	Au (Moz)	Ag (Moz)
Open Pit**					
Indicated	151	0.94	5.0	4.6	24.3
Inferred	16	0.54	2.8	0.3	1.4

*Mineral resources are reported in relation to a conceptual pit shell. Mineral resources are not mineral reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate. All composites have been capped where appropriate.

*Open pit mineral resources are reported at a COG of 0.3 g/t Au. COGs are based on a gold price of USD$1,5500/oz and a gold processing recovery of 88% and a silver price of USD$20/oz and a silver processing recovery of 93%.

*Mining costs were estimated at $1.62/t of total material, processing costs estimated at $15.38/t, and general and administrative (G&A) costs estimated at $1.00/t.

*Pit slope angles ranged from 35 - 50°.

This resource model includes mineralized material in the Camp, East Extension and Portage zones spanning 1,860 m in the southeast direction along the axis of the Portage zone and 900 m in the northeast direction perpendicular to the long axis of the Portage zone. Resource modelling includes mineralized material generally ranging from 340 m to 440 m below surface.

14.13.1 Note on Inferred Resources

An Inferred Mineral Resource is that part of a mineral resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

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, taxation, sociopolitical, marketing, or other relevant issues. The quantity and grade of reported Inferred Mineral Resources in this estimation are uncertain in nature and there has been insufficient exploration to potentially convert some or all of these Inferred Mineral Resources as an Indicated or Measured Mineral Resources and it is uncertain if further exploration will result in upgrading them to the Indicated or Measured Mineral Resource category. SRK is of the opinion that further attempts to convert the remaining Inferred material to Indicated would be of questionable value. The current proportion of the resource classified as Inferred is 10% of total tonnes, and 6% of contained gold. The Mineral Resources in this statement were estimated using the current CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM May 2014).

14.14 Grade Sensitivity Analysis

The Mineral Resources of the Springpole Gold Project are variable depending upon the selected COG. To illustrate this sensitivity, the global block model quantities and grade estimates within the conceptual pit used to constrain the Mineral Resources are presented at different cut-off grades in Table 14-12 for the Indicated Mineral Resource and in Table 14-13 for the Inferred Mineral Resource. The reader is cautioned that the figures presented in this table should not be misconstrued with a Mineral Resource statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of COG. Figure 14-15 presents this sensitivity as grade tonnage curves for the Indicated Mineral Resource and Figure 14-16 displays the same sensitivity curve for the Inferred Mineral Resource.

Table 14-12: Indicated Block Model Quantities and Grade Estimates* at Cut-off Grades

COG Au (g/t)	Quantity (Mt)	Grade Au (g/t)	Grade Ag (g/t)
0.20	172.5	0.86	4.6
0.30	151.0	0.94	5.0
0.40	129.8	1.04	5.4
0.50	109.7	1.15	5.7
0.60	92.2	1.26	6.1
0.70	77.2	1.38	6.4
0.80	64.3	1.51	6.7
1.0	44.6	1.78	7.3

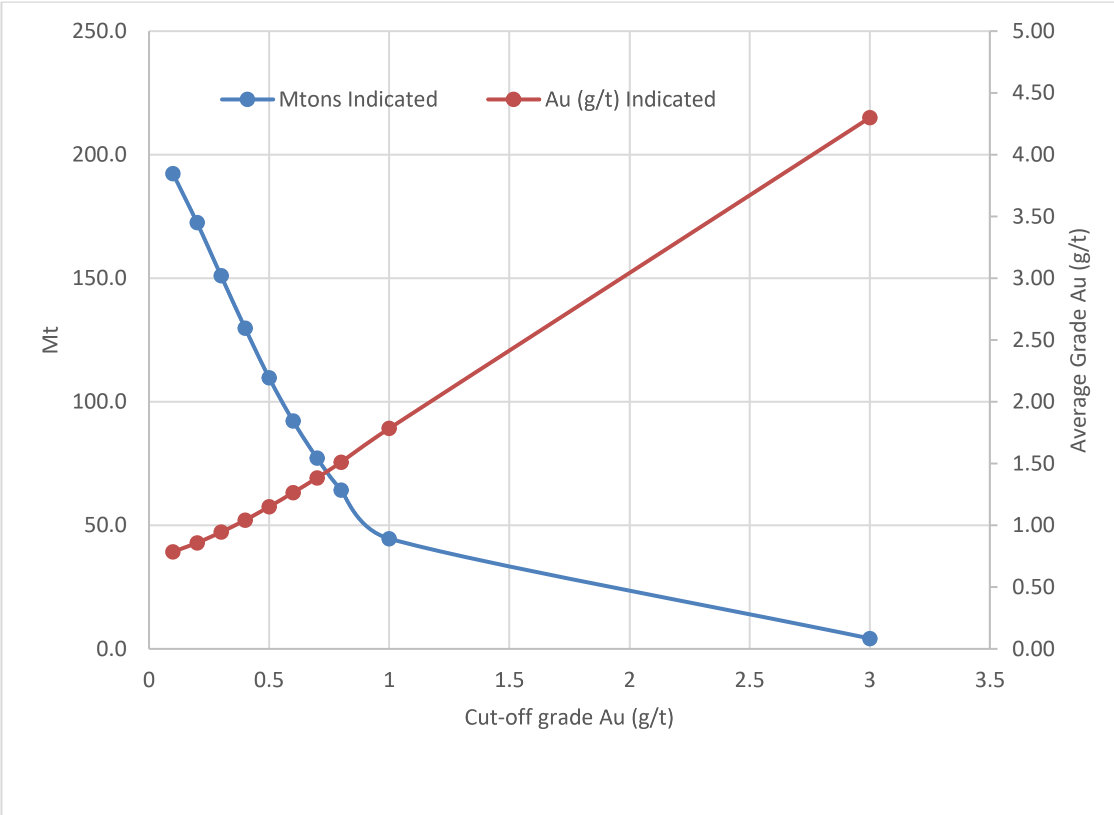
* The reader is cautioned that the figures in this table should not be misconstrued with a mineral resource statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of COG. Mineral Resource base case highlighed in grey.

Table 14-13: Inferred Block Model Quantities and Grade Estimates* at Cut-off Grades

COG Au (g/t)	Quantity (Mt)	Grade Au (g/t)	Grade Ag (g/t)
0.20	20.5	0.48	2.5
0.30	16.1	0.54	2.8
0.40	11.8	0.60	3.0
0.50	7.6	0.69	3.2
0.60	4.4	0.80	3.6
0.70	2.6	0.90	3.9
0.80	1.6	1.01	4.3
1.0	0.5	1.26	5.1

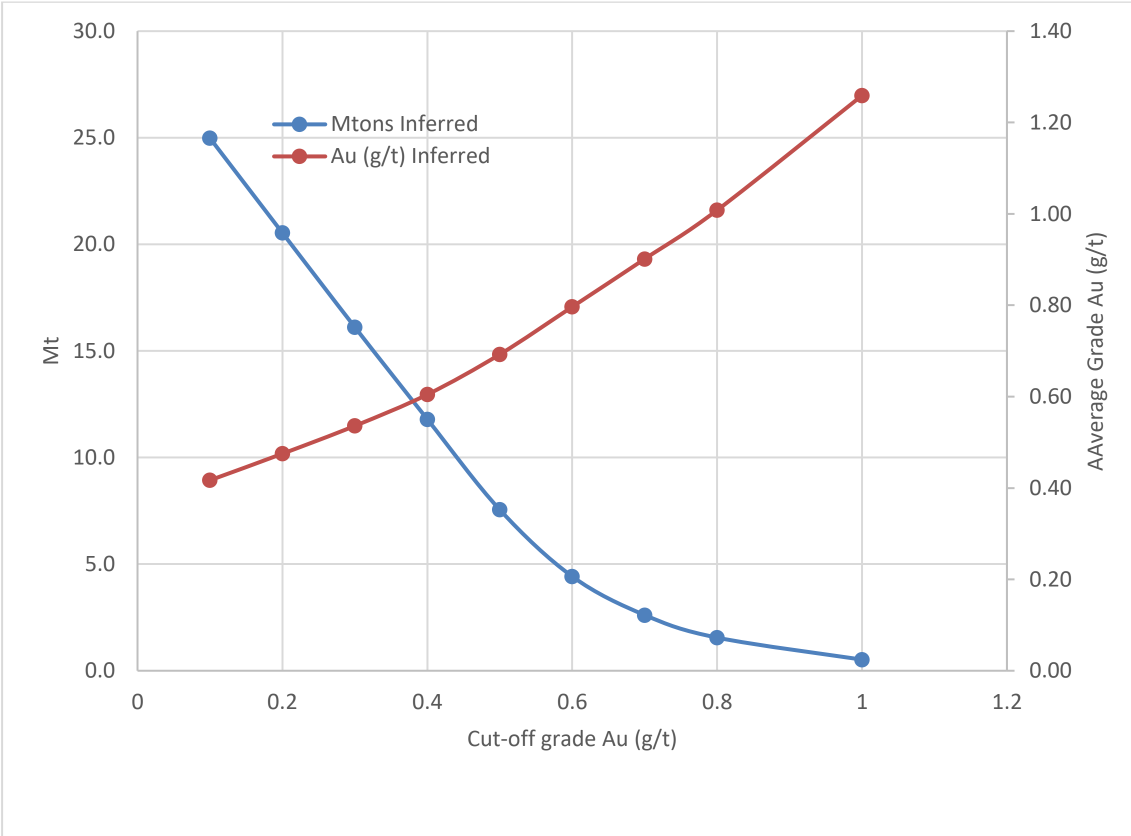
* The reader is cautioned that the figures in this table should not be misconstrued with a mineral resource statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of COG. Mineral Resource base case highlighted in grey.

Figure 14-15: Grade-Tonnage Curves for the Springpole Indicated Mineral Resource



Source: SRK, 2020

Figure 14-16: Grade-Tonnage Curves for the Springpole Inferred Mineral Resource



Source: SRK, 2020

14.15 Previous Mineral Resource Estimates

Mineral resources for the Springpole Gold Project were estimated and reported in a technical report filed on October 15, 2019 (SRK, 2019). This mineral resource included all drill holes for the Project but used a smaller bulk density data set and used different optimization parameters and cut-off to restrict the mineral resource with an open pit resource shell. The mineral resources were reported in accordance with NI 43-101 and are summarized in Table 14-14. These mineral resources are no longer current and are now replaced by the mineral resources presented in Table 14-11 of this report.

Table 14-14: Previous Mineral Resource Statement of September 01, 2019

Classification	Tonnage (Mt)	Au (g/t)	Ag (g/t)	Au Contained (Moz)	Ag Contained (Moz)
Indicated	139.1	1.04	5.4	4.67	24.19
Inferred	11.4	0.63	3.1	0.23	1.12

15 MINERAL RESERVE ESTIMATES

15.1 Summary

The reserves for Springpole are based on the conversion of the Measured and Indicated Mineral Resources within the current Technical Report mine plan. Indicated Mineral Resources in the mine plan were converted directly to Probable Mineral Reserves. There are currently no Measured Mineral Resource estimates and therefore there are no Proven Mineral Reserves. The total reserves for Springpole are shown in Table 15.1.

Table 15-1: Proven and Probable Reserves - Springpole

Category	Tonnes	Grade		Contained Ounces	
	(Mt)	Au (g/t)	Ag (g/t)	Au (Moz)	Ag (Moz)
Proven	0.0	0.00	0.00	0.00	0.0
Probable	121.6	0.97	5.23	3.80	20.5
Total	**121.6**	**0.97**	**5.23**	**3.80**	**20.5**

Note: This mineral reserve estimate is as of December 30, 2020 and is based on the new mineral resource estimate dated July 30, 2020. The mineral reserve calculation was completed under the supervision of Gordon Zurowski, P.Eng of AGP Mining Consultants Inc., who is a Qualified Person as defined under NI 43-101. Mineral reserves are stated within the final design pit based on a USD$878/ounce gold price pit shell with a USD$1,350 /ounce gold price for revenue. The equivalent cut-off grade was 0.34 g/t Au for all pit phases. The mining cost averaged CDN$2.75/tonne mined, processing averages CDN$14.50/tonne milled, and G&A was CDN$1.06/tonne milled. The process recovery for gold averaged 88% and the silver recovery was 93%. The exchange rate assumption applied was CDN$1.30 equal to USD$1.00.

*Pit slope angels ranged from 35 - 50°.

The reserves are based solely on the Springpole open pit.

The QP has not identified any known legal, political, environmental, or other risks that would materially affect the potential development of the Mineral Reserves. The risk of not being able to secure the necessary permits from the government for development and operation of the Project exists but the QP is not aware of any issues that would prevent those permits from being withheld per the normal permitting process.

15.2 Mining Method and Mining Costs

The Springpole Gold Project is amenable to extraction by open pit methods. Preliminary costs were developed based on expected Owner mining. The potential for underground development beneath the open pit has not been examined as part of this Technical Report.

Only Measured and Indicated Mineral Resources were used for the study and all Inferred Mineral Resources were considered to be waste.

This section discusses the development and parameters employed to estimate Mineral Reserves for the current PFS pit design.

15.2.1 Pit Slopes

AGP were retained by First Mining to assist with advancing the mining-geotechnical aspects of the open pit designs for the PFS. First Mining requested that AGP complete a site inspection, and a compilation, review, and assessment of available geotechnical data and information for the Project. The various pit slope design parameters, including geotechnical considerations, are discussed in Section 16 of this Report.

The geotechnical drill hole data was interrogated for preliminary geotechnical domains by SRK, and these have been confirmed by AGP with the additional information from the 2020 drilling program.

Various design sectors were determined for the Springpole pit. Slope stability analyses were undertaken on each sector to determine achievable slope parameters. For all sectors in bedrock, these parameters included the use of a 70° bench face angle, variable berm widths and berms spaced every 24 m vertically. This yielded inter-ramp angles ranging from 38.9 to 54.2°. For the overburden domain, parameters included a 55° bench face angle, a 12.38 m berm and berms spaced every 12 m vertically. No other geotechnical berms were recommended or included in the design.

15.2.2 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 the Project team to select appropriate operating cost parameters for the Springpole open pit.

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 destinations. Process costs and a portion of the G&A costs were based on benchmarking of other relevant studies and test results.

The parameters used are shown in Table 15-2. The revenue 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 226 t trucks using an approximate waste dump configuration to determine incremental hauls for mill feed and waste. The smelting terms and recovery assumptions are based on creating a gold and silver doré.

Table 15-2: Economic Pit Shell Parameters

Description	Units	Value	Metal Prices	
Exchange Rates				
CDN	USD$ =	1.2987		
Metal Prices			**Au**	**Silver**
Price	$/oz		1350.00	20.00
Royalty	%		3%	3%
Net Price Calculation				
Payability	%		99.5%	98.0%
Precious Metal Deduction	$/oz		6.75	0.40
Transportation and Refining	$/oz		5.00	0.00
Subtotal Price	$/oz FOB mine		1338.25	19.60
less Royalty	$/oz FOB mine		40.15	0.59
Net Price	$/oz FOB mine		1298.10	19.01
	$/g FOB mine		41.73	0.61
	CDN$/g FOB mine		54.20	0.79
Metallurgical Information				
Recovery	%		88.0%	93.0%
Power Cost				
Cost of power	CDN$/Kwhr	0.08		
Fuel Cost				
Diesel Fuel Cost to site	CDN$/ l	0.80		
Mining Cost *				
Waste Base Rate - 400m Elevation	CDN$/t	1.67		
Incremental Rate - below	CDN$/t/6m bench	0.0199		
Mill Base Rate - 400m Elevation	CDN$/t	1.62		
Incremental Rate - below	CDN$/t/6m bench	0.0202		
Processing **				
Processing Cost	CDN$/t ore	15.38		
General and Administrative Cost				
G&A Cost	CDN$/t ore	1.00		
Total Process and G&A				
Process + G&A	CDN$/t ore	16.39		

* mining costs based on using 226 t haul trucks
** process costs based on 30 ktpd dry throughput

Wall slopes for pit optimization were based the assessment described in Section 16. For the economic pit shell development, the inclusion of ramps was considered to provide overall slopes of between 35 and 50°. The slope domain locations and parameters are shown in Section 16 with the shallowest slopes occurring in the SW and SE domains.

A series of nested shells were generated using a revenue factor (rf). These were varied between a gold price of USD$135/oz (rf=0.1) and USD$1,620/oz (rf=1.2) to examine the deposit sensitivity to gold prices and outline the higher-value areas. Pit shells were generated at 0.05 rf increments or roughly USD$68/oz increments. This information was graphed, and the various phases and final shell determined based on a net revenue curve.

The final pit is based on the USD$878/oz Au price shell with initial phasing at the USD$473/oz Au shell.

15.2.3 Cut-off Grade

For determining the tonnes and grade in the pit, the marginal cut-off grade was used. The marginal cut-off grade, or milling cut-off, is defined as the minimum grade that would make a profit by processing the material in the mill. This material is already planned to be mined as part of the economic calculations therefore the mining cost is not applicable.

With the cost parameters considered for the Project, this equated to a gold only value of 0.34 g/t. As both gold and silver were recovered for economic value, the net block value after processing was used to determine mill feed or waste for the dilution calculation.

15.2.4 Dilution and Mining Losses

The open pit resource model was provided as an undiluted percentage type model. 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 6 m high. Mining would be completed on 12 m lifts for waste and 6 m lifts for mill feed 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 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.

All mineralized blocks 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 gold and silver grades are zero for dilution purposes.

Comparing the in-situ to the diluted values for the designed final pits, the diluted feed contained 4.9% more tonnes and 3.8% lower gold grade than the in-situ feed summary. The grade dilution percentage is lower than the feed tonnage percentage since the mineralized waste blocks included some grade. AGP considers these dilution percentages to be reasonable considering the nature of the mineralization.

15.2.5 Pit Design

Pit designs were developed for the main pit as well as the small satellite pit immediately to the northeast using a 12 m bench height. The initial pit phase design included only the small satellite pit. This small pit is also above the Springpole Lake level so is convenient from an early mining perspective. The main pit has been divided into phase 2 and 3, with phase 3 being the ultimate pit. The pit optimization shells used to guide the ultimate pit were also used to outline areas of higher value for targeted early mining and phase development. Wall slopes for the inter-ramp were per the final AGP recommendations.

Equipment sizing for ramps and working benches is based on the use of 226 t rigid frame haul trucks. The operating width used for the truck is 8.3 m. This means that single lane access is 27.1 m (2x operating width plus berm and ditch) and double lane widths are 35.4 m (3x operating width plus berm and ditch). Ramp gradients are 10% in the pit and dump for uphill gradients. Working benches were designed for 35 to 40 m minimum mining width on pushbacks.

15.2.6 Mineral Reserves Statement

The reserves for Springpole are based on the conversion of the Indicated Mineral Resources within the current Technical Report mine plan to Probable Mineral Reserves. No Measured Mineral Resources were estimated. These were prepared under the supervision of Gordon Zurowski, P.Eng. of AGP Mining Consultants Inc. who is a Qualified Person as defined under NI 43-101. The reserves are based solely on the Springpole open pit.

This estimate is as of December 30, 2020. The total Probable Mineral Reserves for Springpole are shown in Table 15-3.

Table 15-3: Proven and Probable Reserves - Springpole

Category	Tonnes	Grade		Contained Ounces	
	(Mt)	Au (g/t)	Ag (g/t)	Au (Moz)	Ag (Moz)
Proven	0.0	0.00	0.00	0.00	0.0
Probable	121.6	0.97	5.23	3.80	20.5
Total	**121.6**	**0.97**	**5.23**	**3.80**	**20.5**

16 MINING METHODS

16.1 Overview

AGP was retained by First Mining to prepare a PFS on the Springpole Gold Project. Open pit mining was selected for PFS purposes, based on the size of the resource estimate, grade tenor, grade distribution and proximity to topography. AGP's opinion is that with current metal pricing levels and knowledge of the mineralization, open pit mining offers the most reasonable approach for development. No mining activities have been conducted on this Project to date.

A significant portion of the planned open pit mining area is located below Springpole Lake, so temporary drainage of a portion of the lake will be required while mining is conducted.

16.2 Geological Model Import

The 2019 resource model was provided by SRK on August 8, 2019. This model was used for preliminary mining trade-off studies.

The 2020 resource model was provided by SRK on June 19, 2020. A revision to the classification code was also provided on July 9, 2020. GEMS® software was used for the estimation of resource block model values. SRK provided AGP with resource models in comma separated value (CSV) format for open pit mine planning. The type of block model was a single mineralization percentage model. The grade in each block of the resource model is considered to be an undiluted grade.

Framework details of the different open pit block models are provided in Table 16-1. Resource model item descriptions are shown in Table 16-2 while the final open pit mine planning model items are displayed in Table 16-3. The mining model created by AGP includes additional items for mine planning purposes. Hexagon's MinePlan® software was used for the mining portion of the PFS, using their Lerchs Grossmann (LG) economic pit shell generation, pit, and waste rock storage facility (WSF) design and mine scheduling tools.

The PFS mine plan is based on Indicated Mineral Resources, as no Measured Mineral Resources are contained in the resource model. The block specific gravity (SG) values provided in the resource model were estimated based on new density data, with waste blocks without values receiving a default value of 2.78 t/m^3. Several new model items incorporated since the 2019 PEA include total sulphur and various ARD data estimates.

LIDAR contours and new bathymetry data were received by AGP on January 29, 2020. These two data sources were imported into MinePlan and then merged to create an original ground topography surface for the PFS.

Table 16-1: Open Pit Model Framework

Framework Description	Resource Model Value	Mine Planning Model Value
MineSight® file 10 (control file)	SPRI10.dat	SPRI10.dat
MineSight® file 15 (model file)	spri15.res	spri15.mp3
X origin (m)	548,500	548,500
Y origin (m)	5,692,400	5,692,400
Z origin (m) (max)	-122	-122
Rotation (degrees clockwise)	0	0
Number of blocks in X direction	220	220
Number of blocks in Y direction	210	210
Number of blocks in Z direction	90	90
X block size (m)	10	10
Y block size (m)	10	10
Z block size (m)	6	6

Table 16-2: Resource Model Item Descriptions

Field Name	Min	Max	Precision	Units	Comments
LITHO	0	99	1	-	Rock Type (20=breccia, 50breccia, 50=Metaseds/Andesite/Tuff/waste, 60=Porphyry,70=Trachyte)
DEN	0	5	0.001	t/m^3	Rock density
ORE%	0	100	0.001	%	Percent of the block in mineralized zone (0 to 100)
AU	0	99	0.001	g/t	Au grade in g/t uncapped
AUC	0	99	0.001	g/t	Capped Au in g/t (used for resource estimate)
AG	0	999	0.001	g/t	Silver grade in g/t
AGC	0	999	0.001	g/t	Capped silver grade in g/t (used for resource estimate)
CLASS	0	9	1	-	Resource Class 1= Measured; 2=Indicated; 3= Inferred
ARD	0	9	1	-	PAG/NPAG; PAG=1 NPAG=2
DLNP	0	999	0.01	-	Same as 'Sobek Neutralization Potential'
ICPCT	0	9	0.001	%	Inorganic carbon %
SPCT	0	9	0.001	%	Total sulphur %
SSAP	0	999	0.01	-	Sulphide-sulphur Acid Potential
SNPR	0	9999	0.01	-	Sulphide-Sulphur Net Potential Ratio
SULPH	0	15	0.001	%	Sulphur as sulphide %
ZONE	0	999	1	-	Mineralized zone
SPASS	0	9	1	-	Pass used for estimating sulphur and ARD

Table 16-3: Open Pit Model Item Descriptions

Field Name	Min	Max	Precision	Units	Comments
LITHO	0	99	1	-	Rock Type (0=default,20=breccia,50=Metaseds/Andesite/Tuff/waste,60=Porphyry,70=Trachyte)
DEN	0	5	0.001	t/m3	Rock density
ORE%	0	100	0.001	%	Percent of the block in mineralized zone (0 to 100)
AUC	0	99	0.001	g/t	capped Au in g/t (used for resource estimate)
AGC	0	999	0.001	g/t	capped silver grade in g/t (used for resource estimate)
CLASS	0	9	1	-	Resource Class 1= Measured; 2=Indicated; 3= Inferred, 9=default
ARD	0	9	1	-	PAG/NPAG; PAG=1 NPAG=2, unestimated=0
DLNP	0	999	0.01	-	Same as 'Sobek Neutralization Potential'
ICPCT	0	9	0.001	%	Inorganic carbon %
SPCT	0	9	0.001	%	Total sulphur %
SSAP	0	999	0.01	-	Sulphide-sulphur Acid Potential
SNPR	0	9999	0.01	-	Sulphide-Sulphur Net Potential Ratio
SULPH	0	15	0.001	%	Sulphur as sulphide %
ZONE	0	999	1	-	mineralized zone (see list), previously ROCK in spri15.min
TOPO	0	100	0.01	%	% of block below topography
SLP	0	99	1	-	2020 AGP Slope Domains
OB	1	2	1	-	Overburden flag (1=OB, 2=rock) - defined using SRK 2020 ZONE item
VLT1	0	999	0.01	CDN$/t	Value per tonne - 30ktpd 1350Au MII scenario, July 11, 2020
VLT2	0	999	0.01	CDN$/t	Value per tonne - 30ktpd 1350Au M+I scenario, July 12, 2020
BERM	0	99	0.01	m	berm widths for pit design
VLT3	0	999	0.01	CDN$/t	Value per tonne - 30ktpd 1550Au M+I+I RSC scenario, July 28, 2020
FLAG	0	5	1	-	Dilution flag: 1=mineralized ore, 2=mineralized waste, 3=waste
DILBK	0	4	1	-	number of waste blocks touching an ore block
DILBO	0	4	1	-	number of ore blocks touching a waste block
DORE%	0	100	0.01	%	diluted ore %
DWAS%	0	100	0.01	%	diluted waste %
DAU	0	99	0.001	g/t	diluted Au grade
DAG	0	999	0.001	g/t	diluted silver grade

Global tonnages and grades at various gold cut-off grade values were compared between the original GEMS® format and after import into the MinePlan® software. Very good agreement was achieved as displayed in Table 16-4.

Table 16-4: Global Resource Model Capture

CLASS	Au cut-off	GEMS Model			MinePlan Model			% Difference		
	(g/t)	Tonnage (Mt)	Au (g/t)	Ag (g/t)	Tonnage (Mt)	Au (g/t)	Ag (g/t)	Tonnage (%)	Au (%)	Ag (%)
Indicated	3	4.2	4.30	11.34	4.2	4.30	11.36	0.0%	0.0%	0.2%
	1	45.3	1.78	7.25	45.4	1.78	7.25	0.1%	0.0%	0.0%
	0.8	65.6	1.50	6.66	65.7	1.50	6.66	0.1%	0.0%	0.0%
	0.7	79.2	1.37	6.32	79.3	1.37	6.32	0.1%	0.0%	0.0%
	0.6	95.7	1.25	5.96	95.8	1.25	5.96	0.1%	0.0%	0.0%
	0.5	115.1	1.13	5.60	115.3	1.13	5.60	0.1%	0.1%	0.0%
	0.4	138.0	1.02	5.22	138.2	1.02	5.22	0.1%	0.1%	0.0%
	0.2	190.0	0.82	4.39	190.2	0.82	4.39	0.1%	0.1%	0.1%
	0.1	216.1	0.74	4.01	216.2	0.74	4.01	0.1%	0.0%	0.0%
	0	229.4	0.70	3.82	229.4	0.70	3.82	0.0%	0.0%	0.0%
Inferred	3	0.0	0.00	0.00						
	1	0.6	1.25	4.62	0.6	1.24	4.62	0.6%	0.1%	0.1%
	0.8	1.8	1.01	4.02	1.8	1.00	4.00	0.5%	0.1%	0.3%
	0.7	3.1	0.89	3.70	3.1	0.89	3.69	0.1%	0.0%	0.1%
	0.6	5.6	0.78	3.31	5.6	0.78	3.31	0.2%	0.1%	0.0%
	0.5	9.8	0.68	3.06	9.8	0.68	3.06	0.3%	-0. %	0.1%
	0.4	16.5	0.59	2.85	16.5	0.59	2.85	0.2%	0.1%	0.0%
	0.2	33.7	0.44	2.26	33.7	0.44	2.26	0.1%	0.0%	0.0%
	0.1	40.1	0.39	2.05	40.2	0.39	2.05	0.1%	0.0%	0.0%
	0	48.9	0.33	1.73	48.9	0.33	1.73	0.0%	0.0%	0.0%

16.3 Open Pit Mining Geotechnics

AGP were retained by First Mining to assist with advancing the mining-geotechnical aspects of the open pit designs for the PFS. First Mining requested that AGP complete a site inspection, and a compilation, review, and assessment of available geotechnical data and information for the Project.

AGP's J. R. Tosney, P.Eng. visited the Project site from August 31 to September 1, 2020. During the visit Mr. Tosney completed the following tasks:

- met with First Mining geology and exploration staff to discuss and review the Project, and the ongoing exploration and geotechnical drill plan and status
- reviewed local and regional geology reports, plans and sections
- collected and compiled available site geological and geotechnical data
- completed domain-scale geotechnical logging for select intervals of drill core available at the time of inspection

- completed boat and on-foot traversing to rock outcrops and drill sites and conducted geotechnical mapping and rock mass characterization of constituent rock masses; data collection tasks focused on verifying and supplementing existing information, including lithology, rock mass strength, and discontinuity characteristics

Initial estimates of suitable pit slope angles were developed by SRK (2017, 2019) and these have been reviewed and confirmed by AGP for current PFS-level mine planning. The pit slope assessment is based primarily on resource and geotechnical drilling data and core photographs, simple rock quality designation (RQD) and rock mass classifications data, Whittle pit shells, geologic models, and relevant background reports.

16.3.1 Geological Setting

Per SRK (2019), a polyphase alkali, trachyte intrusion displaying autolithic breccia textures lies at the heart of the Springpole Gold Project. The intrusion is comprised of a system of multiple phases of trachyte believed to be part of the roof zone of a larger syenite intrusion. Recent exploration has confirmed trachyte intrusive rock is the dominant lithology within the Project area and is a host to mineralization. Interpretation of the intrusive complex is complicated by a mixture of overprinted regional and local metamorphic events related to burial and tectonism.

Pervasive alteration and metamorphism have reduced the original porphyry intrusion to a complex alteration assemblage dominated by sericite, biotite, pyrite, calcite/dolomite, and quartz. Primary igneous textures are well preserved in places and give indications to the possible genesis of the initial phase of gold mineralization.

All rocks on the property exhibit pervasive alteration that consists of multiple overprinted phases. Regional metamorphism has subsequently altered the primary hydrothermal mineral assemblages, but textures have been preserved with the exception of areas of high strain (e.g. northwest trending shear zones). Advanced argillic alteration appears throughout the trachyte intrusion and occurs in some of the late-stage lamprophyre dikes, though on a small scale. It is difficult to assess at what stage argillic alteration occurs, but it appears to define an envelope around the Portage zone potassic-alteration/mineralization.

16.3.2 Structural Geology

Deformation has added complexity to the apparent geometry of, and the potential of, the Springpole Gold deposit. Gravity and magnetic surveys carried out across the Project demonstrate that several phases of deformation are evident. Banded iron formations describe north-northwest facing tight to isoclinal antiforms and synforms.

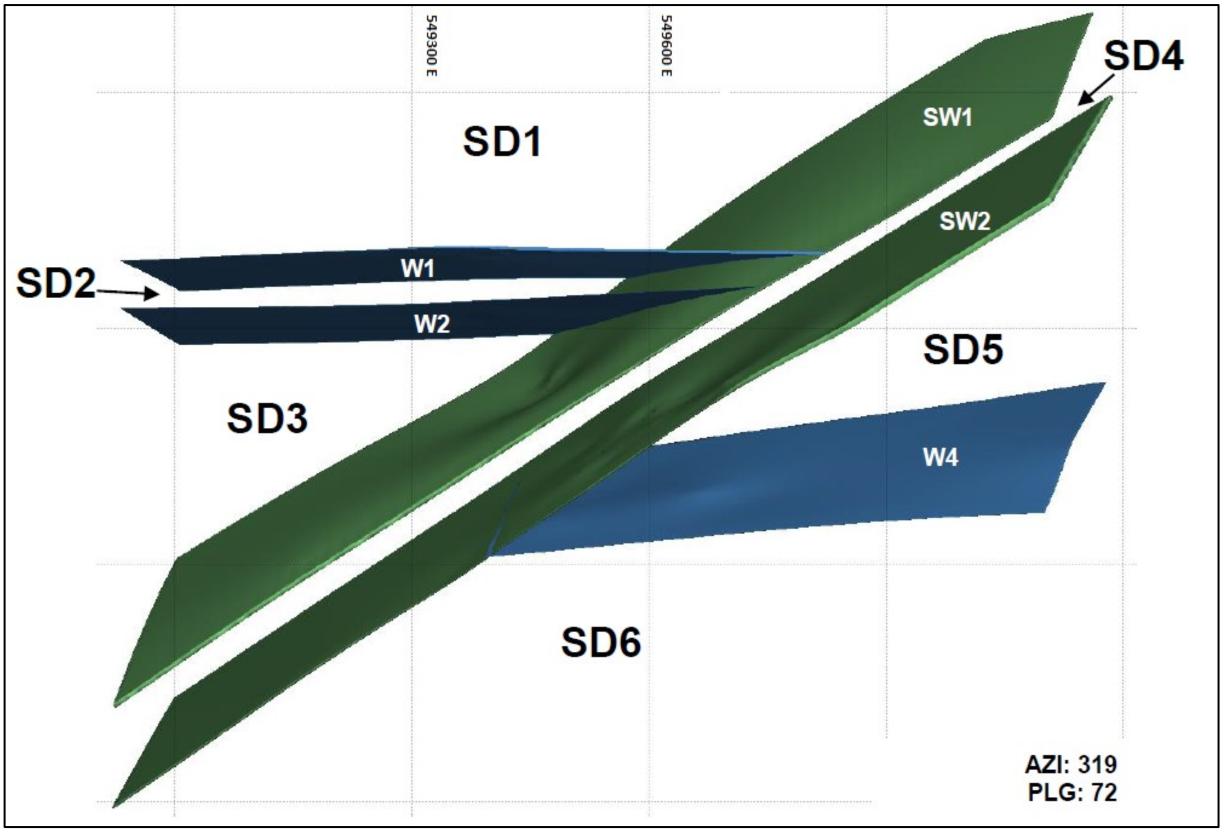
In 2011, SRK was contracted to carry out a preliminary study of the structural controls on the mineralized deposit geometry. The study found that the deposit was subjected to several deformational events resulting in the structural domaining illustrated in Figures 16-1 and Figure 16-2, including, but not limited to:

- early folding resulting in tight to isoclinal fold geometries and development of associated shear zones
- intermediate large-scale, potentially deep-rooted shear zones

- late stage brittle faulting

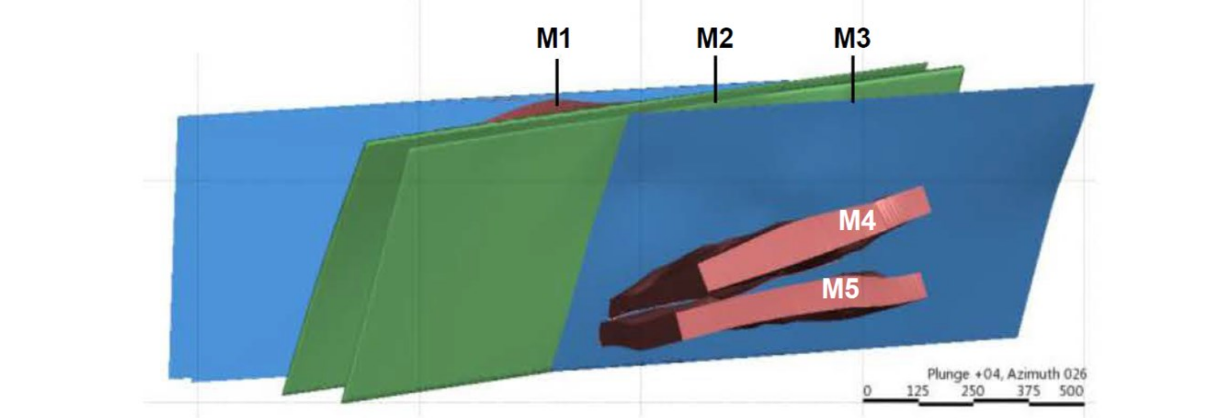
Continued intense deformation and associated metamorphism manifesting as folding, strike-slip faulting and shearing, coupled with regional green schist metamorphism of the region obscures primary textures and likely leads to some (possibly minor) degree of precious metal remobilization.

Figure 16-1: Top View (azimuth 319, plunge 72) of Structural Domains SD1-SD6



Source: SRK, 2011

Figure 16-2: Springpole Preliminary Structural Geology Model (SRK, 2013)



Side view (azimuth 026, plunge 04) of southern shallowly dipping weak zones.

Source: SRK, 2013

16.3.3 2013 Geotechnical Investigations

A 2013 drill program was designed and executed by SRK to better understand the rock mass characteristics and enable slope design and stability analysis at an appropriate level of study.

Per SRK, the 2013 oriented core program included the drilling and logging of six HQ-3 diameter holes (SG13-200 to SG13-206) from which intact rock strength laboratory samples were collected for analysis. A structural geology model was developed for the Project and SRK issued a technical memo focused on identifying structures and zones that may impact geotechnical and hydrogeological conditions and mining outcomes (SRK, 2014).

Faults within the proposed Springpole open pits and immediate vicinity have been modelled. To aid the development of the preliminary 3D structural model (and to gain a preliminary understanding of the possible fault trends and their influence on the proposed pit) the topography and bathymetry in the form of a digital elevation model was interpreted for lineaments and zones of weakness possibly associated with rock mass damage in fault zones. This information assisted with the preliminary domain delineation resulting in six structural domains, identified in Section 16.3.5.

16.3.4 2020 Geotechnical and Hydrogeological Investigations

As part of the 2020 drilling program, First Mining completed a follow-up campaign of geotechnical and hydrogeological drilling, logging, testing, and sampling (Fracflow, 2020).

Per Fracflow (2020), the main objectives of the 2020 combined geotechnical-hydrogeological program ("SGH2020") were 1) to extract fracture geometry data from representative zones or zones with similar geometry, fracture density, etc., around the proposed open pit perimeter; 2) to measure hydraulic conductivity values from the fractured rock mass along the pit wall of the proposed main open pit to improve estimates of mine water pit inflows and pit dewatering; 3) to measure rock properties using the drilled cores from representative zones; and 4) to collect fracture geometry data and hydraulic conductivity values at the two cofferdam areas.

The SGH2020 program consisted of drilling HQ inclined boreholes, core logging of drilled cores, packer tests, fracture surveys using acoustic televiewer, rock testing (point load tests and Brazilian tests), and multi-level piezometer installation. The bearing and plunge of the boreholes for the SGH2020 drill program were selected to determine rock properties and fracture orientations around the pit perimeter. Most of the SGH2020 boreholes had a southeast bearing that supplemented and reduced the bias in the fracture orientation data that had been created in 2019, since only exploration boreholes were available for packer testing and acoustic televiewer surveys in 2019 and those boreholes generally had a north-east to south-west bearing.

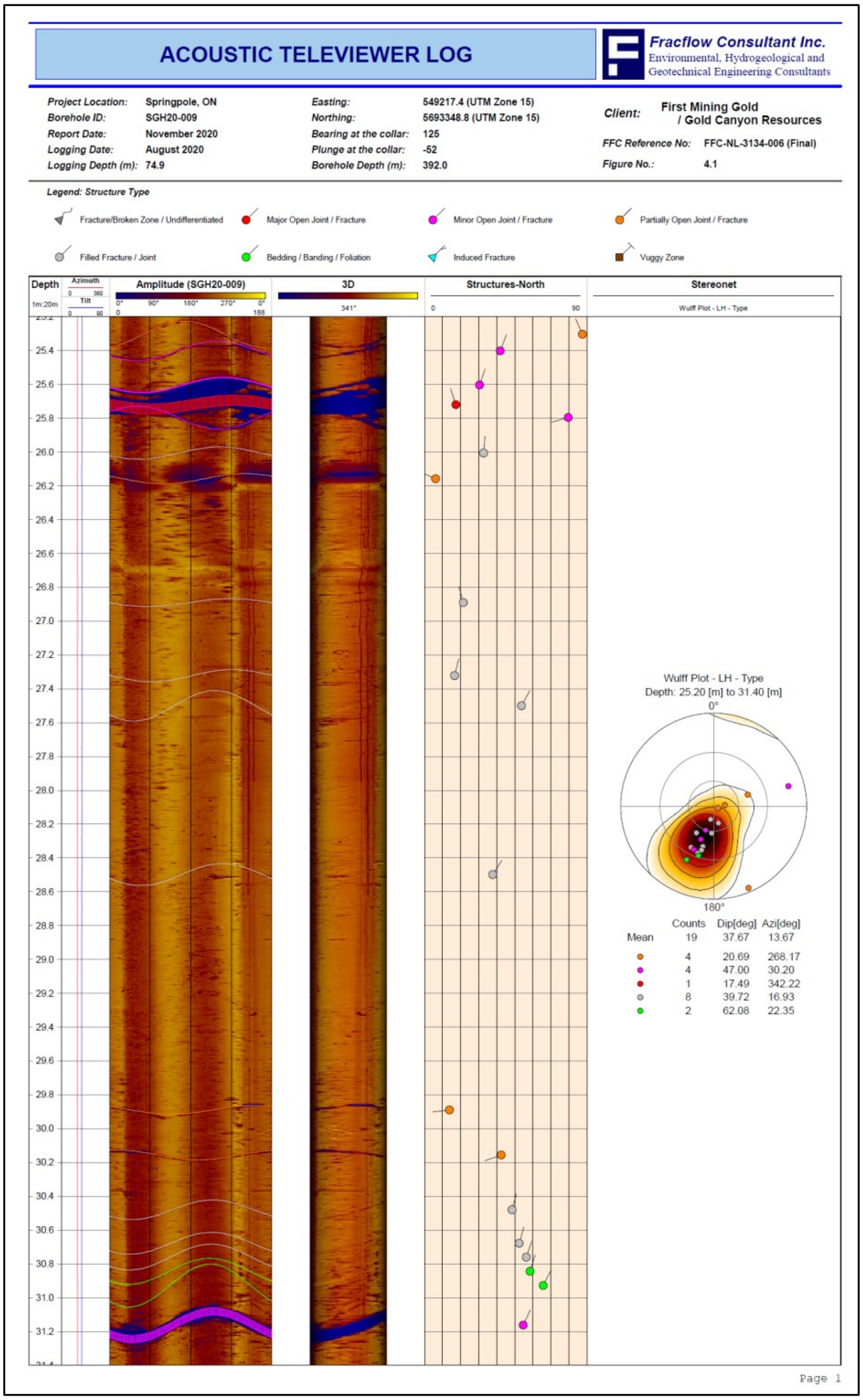
Ten boreholes were completed, eight inclined boreholes along the pit wall perimeter of the proposed main open pit and two inclined boreholes at the two cofferdam areas. In addition, one vertical borehole for characterizing the aquifer was completed at the north side of the current camp zone. The plunge of the inclined boreholes ranged between -50 and -53° with depths or lengths that ranged between 293 and 440 m.

Cores recovered from the boreholes were delivered to the camp for general core logging. The geology intersected by each borehole was mapped by First Mining staff and the basic core properties such as percent core recovery and RQD were also recorded by First Mining staff. Fracflow staff focused primarily on characterizing the geometrical and mechanical properties of the fracture system and developing searchable core photograph files.

Packer tests were conducted in all of the SGH2020 boreholes. All packer tests were conducted using the end of the borehole approach. Generally, the packer tests were completed in two stages, first when the borehole had reached approximately one half its proposed length and a second stage when the borehole had reached its planned depth.

All of the SGH2020 boreholes were logged over their open hole section using an acoustic televiewer (AT) after the packer tests had been completed. A typical AT log is presented in Figure 16-3. Approximately 3,278 m was logged in the 11 boreholes that were drilled around the main proposed open pit using the acoustic televiewer. The AT survey managed to log to near the bottom of six boreholes, SGH20-002 SGH20-004, SGH20-005, SGH20-006, SGH20-007, and SGH20-008, which showed moderately fractured cores. However, in the other five boreholes, the AT survey was terminated by either borehole blockage or when large open fault zones were encountered. Fracture frequency per metre for those 11 boreholes ranged between 0.6 and 2.6.

Figure 16-3: Typical Acoustic Televiewer (AT) Log - SGH20-009



Source: Fracflow, 2020

A single piezometer interval was installed in each of three boreholes, SGH20-001, SGH20-002 and SGH20-004 and two-level or two interval piezometers in SGH20-008.

Rock property tests (point load tests and Brazilian tests) were completed in the 10 inclined boreholes and the vertical borehole, SPW20-001.

16.3.5 Slope Design Sectors and Criteria

Per SRK (2019), and confirmed by AGP for the current study, individual drill logs have been recorded for rock drilling on the Project, and they include lithological and alteration descriptions of major lithologies. Total core recovery and rock quality designation has been relatively consistently and correctly recorded for most of the drilling evaluated for the Project. Empirical rock strength estimates are recorded for more recent (2011 onward) drill core recovered. Unconfined strength point load testing, and joint-condition data were acquired from the 2013 and 2020 oriented-core programs.

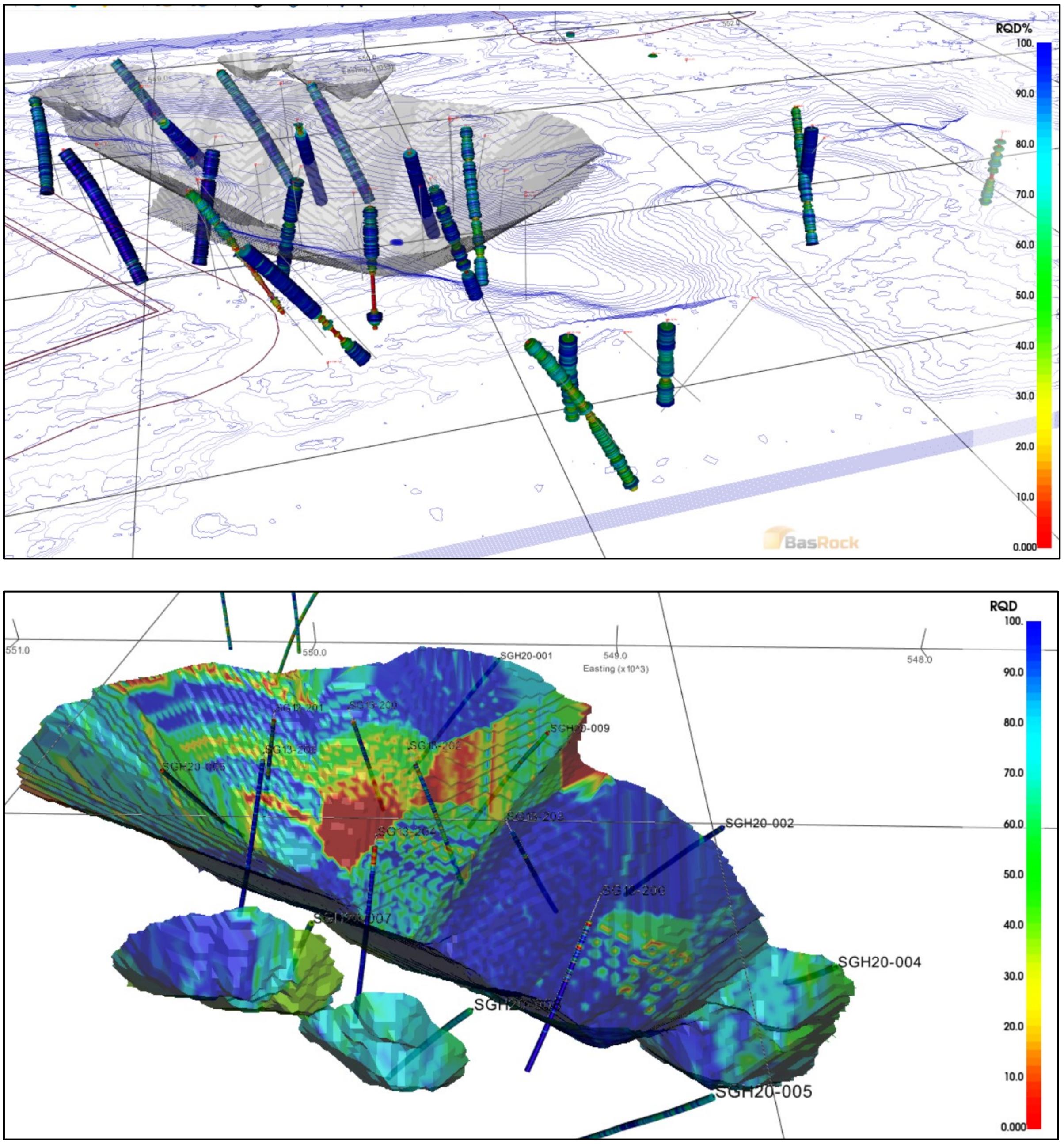
Geotechnical drill hole targets have been selected to represent the rock mass likely to be encountered in the proposed pit walls. The core-box photographs, in conjunction with all available logs, were viewed and rated for solid screen recovery, geological strength index (GSI), and intact rock strength (IRS). Depictions of geotechnical drill hole coverage and RQD measurements, as of December 2020, are presented in Figure 16-4.

The data were interrogated for preliminary geotechnical domains by SRK, and these have been confirmed by AGP. Below the lake-floor sediments and glacial overburden, there are at least three distinct rock mass domains within the pit +200 m envelope. A Strong-domain, which surrounds Springpole Lake, is considered the host or “country-rock”. There is a relatively narrow transition into the Intermediate-domain and Weak-domain. The Intermediate-domain may be related to regional-scale faulting in some areas but is also the transition towards the centre of mineralization. The Weak-domain appears to be directly associated, spatially, with the mineralization. Strong rocks dominate in the north, and on the southwestern flank of the pit. The Weak-domain appears continuous through the centre of the pit, with the southern-most slopes likely being composed of these rocks. The three domains’ intact rock strengths and geological strength indices are considered to be in these ranges:

- strong-domain, IRS ≈ 150 ±50 MPa, GSI ≈ 60 ±30
- intermediate-domain, IRS ≈ 75 ±35 MPa, GSI ≈ 40 ±25
- weak-domain, IRS ≈ 30 ±20 MPa, GSI ≈ 25 ±20

Overall, the data indicates generally ‘fair’ to ’good’ rock mass conditions throughout the majority of the mining zone (i.e. the ‘general/mean’ geotechnical unit) with poorer quality rock masses and local bench-scale slope instabilities likely to be encountered in the Weak-domain, and in zones proximal to fault intercepts, and adjacent to fault zones.

Figure 16-4: Geotechnical Drill Hole Coverage and RQD measurements, as of Dec 2020



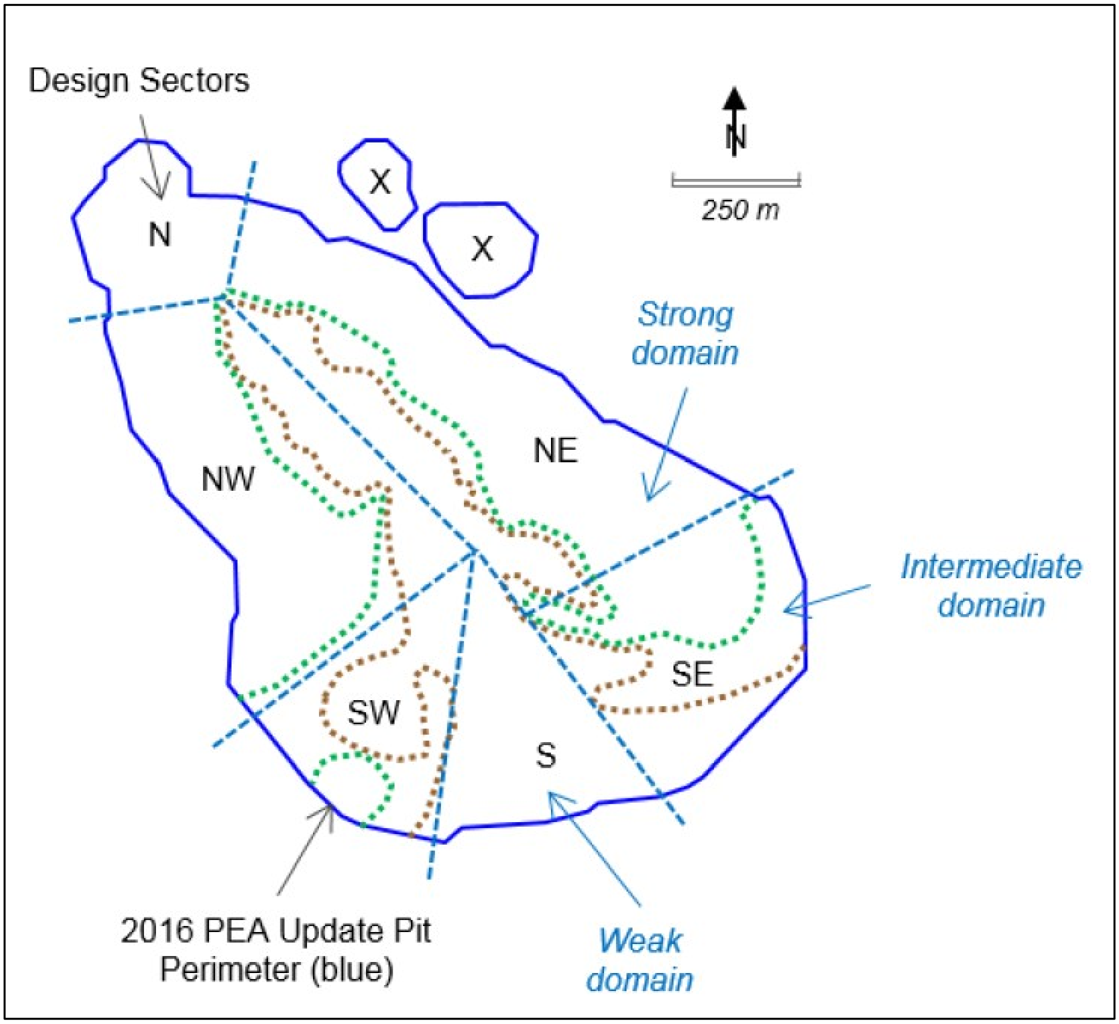
Source: AGP, 2021

Per SRK (2019), the overall trend is strong (Archean-aged basement) rock in the north, strong to intermediate strength rock on the southwest and northeastern flanks of the proposed pit, and weak mineralized (mafic) rock at depth, along the pit mid-line, and in the south. Alteration fluids introduced during the mineralizing events appear to have weakened the intact rocks in the immediate vicinity of higher-grade zones. The alteration "halo" is approximately 50 m wide.

Recent geotechnical data available since the 2017 PEA generally confirms the model and previous geotechnical data. Hence, the pit shape remains partitioned into the same preliminary design sectors as defined by SRK (Figure 16-5).

SRK and AGP have both used rock mass domains to determine the slope composition and to estimate an overall rock mass rating for each section, as summarized in Table 16-5. These estimated overall-slope values, taking into consideration the weaker rocks in the toe of most of the slopes, were compared to published design charts, and modeled conceptually in 2D and 3D, to estimate achievable overall slope angles for a factor of safety of at least 1.2.

Figure 16-5: Location of Slope Domains



Source: (SRK, 2019)

Table 16-5: Overall Slopes for Economic Pit Shells

Design Sector	Zone Code	Inter-Ramp Angle (degrees)	Haul Roads In Slope	Slope Height (m)	Overall Slope (degrees)	Rock Mass Rating (estimate)
OB	1	30.0	0	12	30	
N	2	54.2	1	300	50	good
E	3	51.0	2	375	45	fair
SE-n	4	38.9	2	375	35	poor to extremely poor
SE-s	5	44.9	2	375	40	fair to poor
SW	6	38.9	2	375	35	extremely poor to poor
W	7	51.6	2	340	45	fair to poor

Note: 35.4 m haul road width

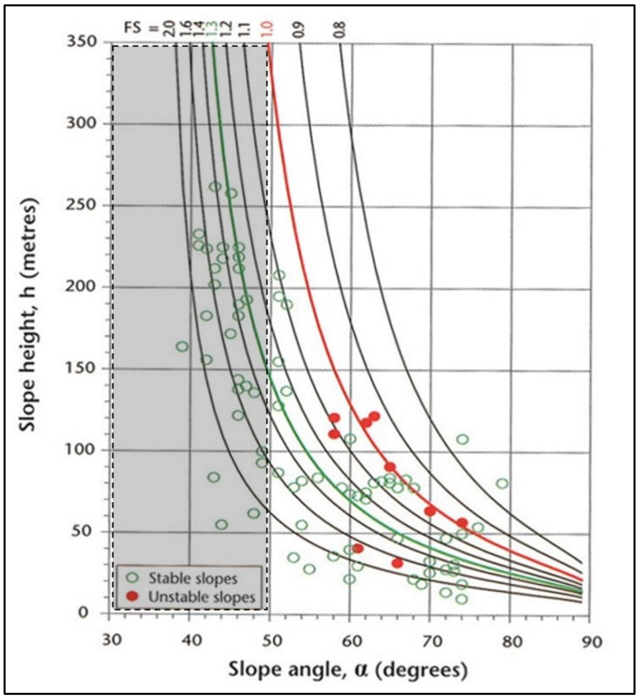
There is possibly an up-side opportunity for steeper slope angles used for design if a significant increase in the confidence of the geotechnical data (and model) can be achieved. What may ultimately control achievable slope angles (apart from hydrogeological constraints) is the Weak to Intermediate-domain spatial arrangement, and anisotropy in the host rock in the Strong-domain.

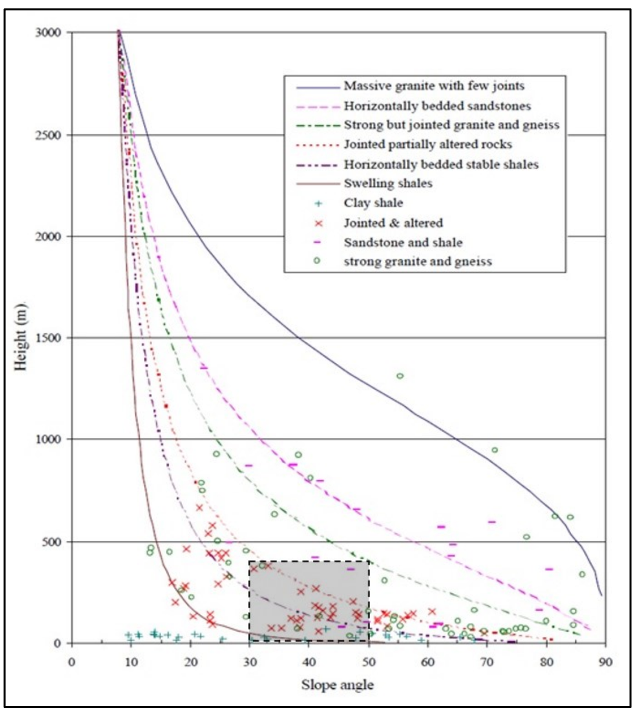
AGP notes efforts to 'maximize' slope angles based on limited data can often lead to overly optimistic designs and related project economics. These can be difficult to 'walk back' if, or when, contrarian data is recorded during subsequent investigation work. AGP recognizes the potential to improve upon/optimize relatively conservative initial guidance, if/when additional confirmatory data becomes available.

16.3.6 Preliminary Open Pit Slope Stability Assessment

The following widely used empirical slope stability charts (Figure 16-6) demonstrate typical safety factors for a variety of slope configurations and rock types and include AGP's PFS guidance for the Project.

Figure 16-6: Empirical Slope Stability Charts, with Springpole Design Range Indicated in Shaded Areas





Source: AGP, 2021

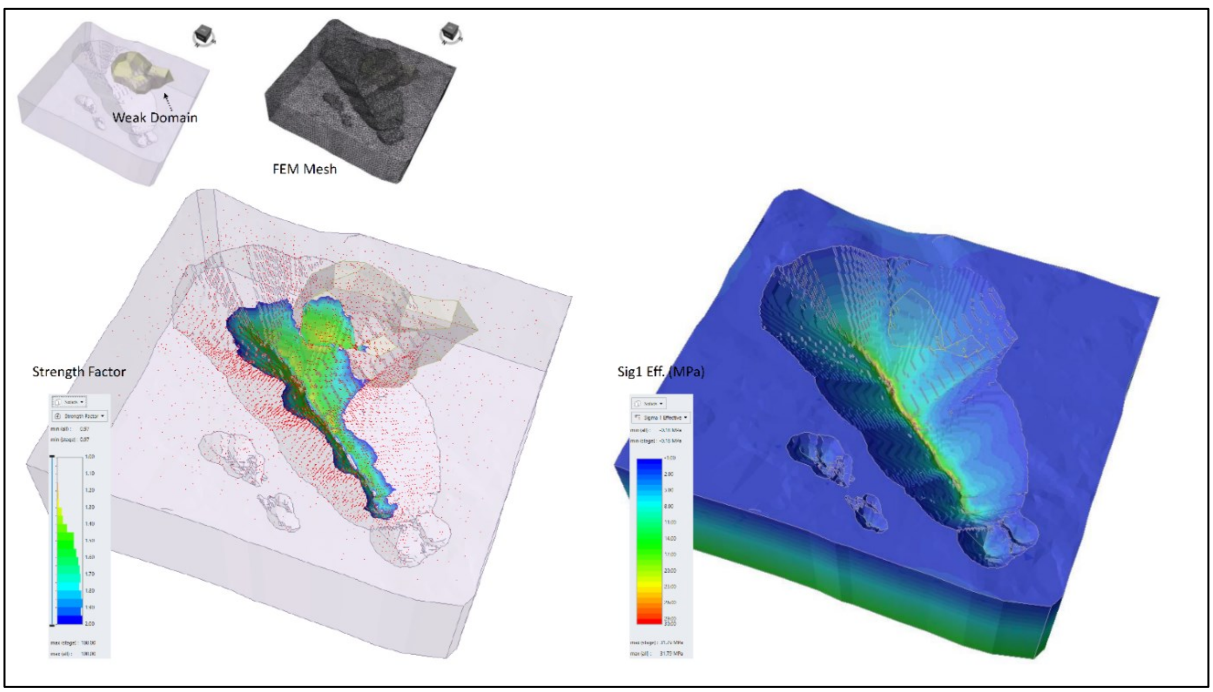
Preliminary 2D limit equilibrium and 3D Finite Element stability and stress analyses have been completed for the Project by AGP using the Mineral Resource pit shell and the initial pit design guidance described in the previous sub-sections, to gather initial insight into inter-ramp and overall slope 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, and ground water conditions, as well as the effects of excavation sequencing.

AGP commonly uses the following approach for target factor of safety (FOS/strength factor) values at the PFS level:

- 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.

For the current assessment slope heights ranging from 100 to 375 m with inter-ramp and global slope angles varying from 30° to 50° were analyzed under fully to partially saturated conditions. Typical analyses results are presented in Figure 16-7.

Figure 16-7: Typical Results for 3D FEM Stress Analyses



Source: AGP, 2021

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 for the current study due to insufficient discontinuity and orientation data; bench configurations have included an allowance for reasonable catchment widths to help manage operational challenges that may arise from local bench-scale stabilities.

It is probable that unfavorably oriented geological structures are locally present within various slope pit sectors, particularly given the size and extents of the pit and the observed variability in discontinuity orientations. 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.

On the other hand, both (unlikely) seismic loading and multi-bench-scale to pit-scale structures have the potential to significantly affect overall pit slope stability. The current status and impact of these remain largely unknown. Per SRK (2019), and confirmed by AGP, the Springpole Gold Project is located within a low seismic hazard zone.

The inclusion of hypothetical adversely oriented faults and bedding planes in the stability analyses indicates potential FOS's less than 1.0, particularly with (low probability) seismic loading applied. Further geotechnical investigations are warranted to determine the location and character of inter-ramp to global-scale features and poor rock quality zones that may impact stability and mining outcomes.

Sufficient data have been compiled regarding geotechnical strengths and characteristics of the primary rock types to provide a range of potential pit wall design guidelines. 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 should be generated and analyzed as updates to the mine plan and/or geotechnical domain model become available.

16.3.7 Geotechnical Model Limitations

The preceding section summarizes information and knowledge gathered to date, primarily by others, along with information collected by AGP during a two day site visit in 2020.

This information provides the basis for preliminary pit slope design and guidelines to assist with mine design, 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. Where appropriate, geological features identified should be verified and validated with additional field work and interpretation.

16.3.8 Data Gap Analysis

A geotechnical data gap analysis has been completed by AGP to determine data requirements to support a Feasibility Study (FS) level mine design for the proposed open pit (Table 16-6).

The available data were 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 sectors of the open pit mine
- Geological Coverage - ensuring sufficient characterization of the different geological units (lithologies) expected to be in and around the open pits.
- 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 at each deposit.

Table 16-6: Gap Analysis

Gap Analysis Criteria	Status	Gaps
Spatial Coverage	Fair - Good	- A fair amount of geotechnical data has been collected to date within the pit extents; however, drill holes do not intersect large portions of the proposed pit walls. - Ongoing "basic" and "detailed" geotechnical data, and structural data collection is required is for all slope sectors
Geological Coverage	Fair - Good	-A significant amount of geological data and limited geotechnical data currently exists for the Project area and defined geotechnical domains. -Ongoing "basic" and "detailed" geotechnical data, and structural data collection is required is for all geotechnical domains
Coverage of Major Features	Fair	-Initial fault characterization work based on core data, more work and interpretation required with 2020 geotechnical data. -Limited spatial and geotechnical knowledge exists regarding location and intensity of fault impacted zones. Limited orientation and persistence data available
Orientation Data Bias	Good	-Ongoing orientation data collection and analysis required (evaluated using coring data and / or ATV/OTV surveys)
Orientation Data Quality	Fair	-Ongoing orientation data collection and analysis required (evaluated using coring data and / or ATV/OTV surveys)
Field and Laboratory Strength Testing	Limited	-Limited rock estimates, mainly from field index testing -Uniaxial compressive strength (UCS), tri-axial, tensile, direct shear and other standard laboratory tests are required to determine / confirm rock strength & deformation parameters, discontinuity strength criteria.

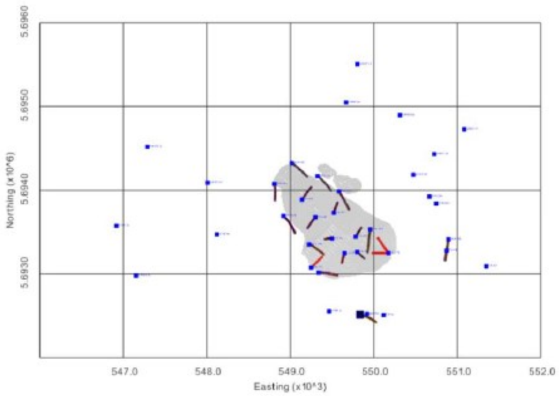
The results of the geotechnical gap analysis indicate a number of important factors that require additional investigation. For Feasibility designs, a higher level of confidence is required, and the preliminary geotechnical model presented will need to be updated with additional data and perhaps more importantly, additional integrated structural geotechnical interpretation and analyses. Recommended data collection and interpretation tasks are outlined below. These recommendations could potentially be completed in phases, using combinations of third-party consultants and First Mining staff.

The following geotechnical work is recommended to advance the Project to an FS level:

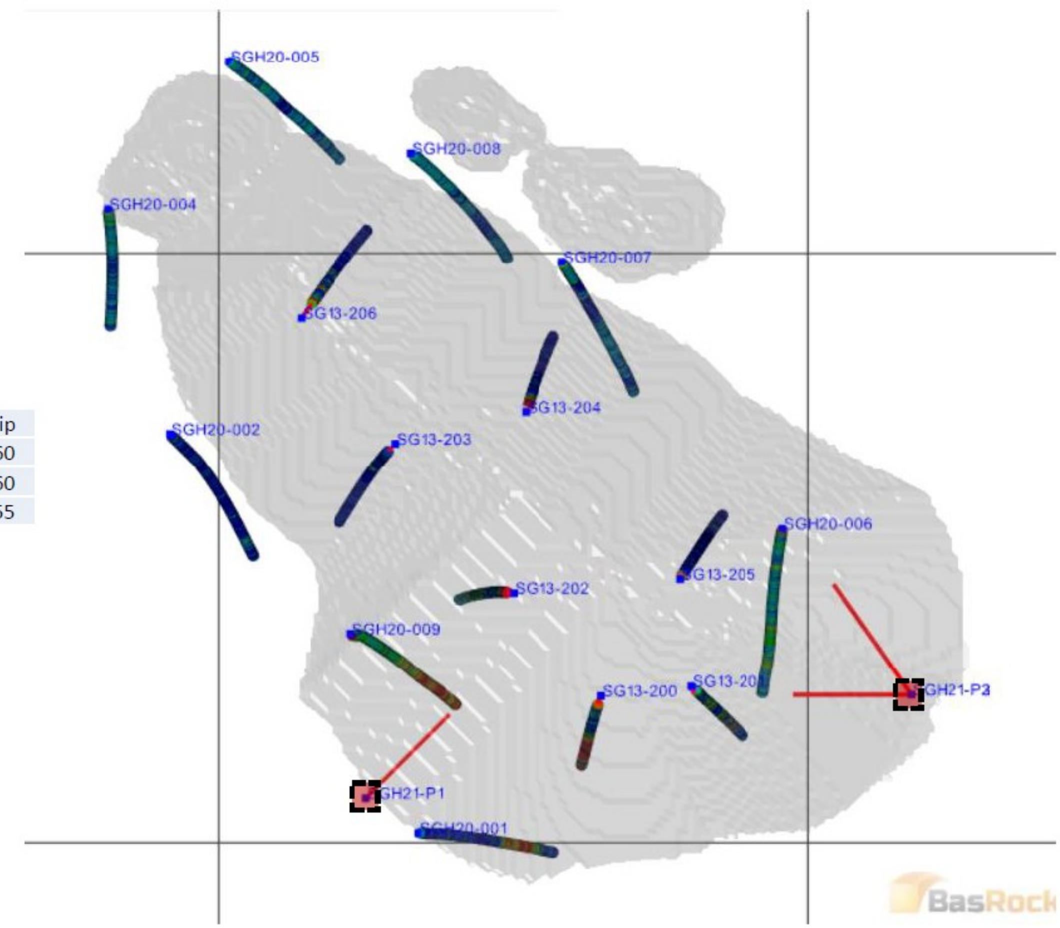
Geotechnical Drilling and Core Logging

- A 3-hole geotechnical drilling and rock mass characterization program (Figure 16-8) is proposed to achieve FS data requirements, including targeted drilling of current data voids and areas of interest in the SW-S-SE sectors, to include discontinuity orientation measurements (where possible), sampling for laboratory strength testing, and televiewer surveys. The holes mainly target waste rock zones outside of the ore zone to determine the geotechnical properties of the units forming the pit walls. Planned metallurgy and infill holes in other areas of the pit can be used as dual purpose holes for collecting additional “quick / basic” geotechnical data. The core holes should be drilled using a triple tube core barrel to preserve the integrity of the core while drilling and retrieving.

Figure 16-8: Proposed Geotechnical Drillholes



ID	Easting	Northing	Elev	Depth	Azimuth	Dip
SGH21-P1	549250	5693075	400	400	45	-60
SGH21-P2	550175	5693250	400	400	270	-60
SGH21-P3	550175	5693250	400	400	325	-55



Proposed Pit Geotech (SGH21-P01, P02, P03)

Source: AGP, 2021

- Laboratory testing is also recommended, including uniaxial compressive strength testing (with strain measurements), tri-axial strength testing, direct shear testing of discontinuities, and index testing of discontinuity infill materials. Results may be used to confirm or update the geotechnical analysis and slope design parameters provided. Samples should be collected from dedicated (or first priority) geotechnical drill holes to ensure the appropriate materials are sampled, and to avoid conflicts with exploration sampling and assaying requirements. UCS and Triaxial testing should be completed for each of the significant lithological units. The triaxial testing should focus on characterizing the intact rock strength both across and parallel to foliation. Samples of fault or dike contact gouge should also be collected and tested to help characterize the strength of these materials.
- Include the preliminary 3D lithostructural interpretation of all major faults, as viewed in drill core and rock outcrop within 200 m of the pit crest, and integrate them with the regional structural interpretation, into an exploration and geotechnical model for the FS.
- Produce robust 3D digital wireframe models of lithology, alteration with intensity, and structures.
- Update / confirm 2D and 3D stability models and assessments with the aim of optimizing pit slope designs according to anticipated geotechnical performance characteristics.

16.4 Hydrogeological Considerations

Prior to 2013, there was no collection of hydrogeological data from the Springpole site. In early 2013, SRK initiated a Pre-Feasibility level geotechnical and hydrogeological data collection program for the open pit. This field program concluded in March 2013 and included 20 packer tests in seven core holes drilled within the proposed pit footprint. A full interpretation of the data was not available for the previous PEA reporting as the PFS-level work was put on hold, so results are considered to be at draft level. However, hydrogeological observations from site and preliminary testing data were used for this Report to inform the PFS-level assessment of the probable hydrogeological conditions at the site .

Additional groundwater investigations were carried out by North Rock Environmental in 2017 and 2018 (North Rock, 2018, 2019). This program was designed to provide a groundwater monitoring system across the site and initiate long-term site-wide data collection. Monitoring wells were installed in five overburden sites (MW1 to MW5) and in six deeper exploration boreholes (DDH1, DDH2, BL-235, BL08-235, SP11-102, and SP11-064) located across the study area. Groundwater levels and samples were obtained from these wells and form the basis of a baseline hydrogeochemical data base.

Fracflow also completed a series of packer testing during the 2020 investigation program. Packer tests were conducted in all of the SGH boreholes. For example, for the five boreholes located at the west side of the open pit the packer tests included 24 rod tests and 48 injection tests with 119 pressure injection steps. All packer tests were conducted using the end of the borehole approach. The hydraulic conductivity values were calculated for each test interval and for those five boreholes ranged between 4.0 E-10 m/s and 1.8 E-5 m/s. Generally, the packer tests were completed in two stages, first when the borehole had reached approximately one half its proposed length and a second stage when the borehole had reached its planned depth.

Hydraulic conductivity values for discrete intervals were estimated using the packer test data from the SGH2020 program. Multi-level monitoring wells were installed in four boreholes, SGH20-001, SGH20-002, SGH20- 004 and SGH20-008, after drilling and acoustic televiewer surveys. Placement of the screened portion of each well was based on the drilled cores and the acoustic televiewer survey data collected from the corresponding boreholes. A single (deep) piezometer was installed in SGH20-001, SGH20-002 and SGH20-004, and a multi-level piezometer (shallow and deep) was installed in SGH20-008.

Per SRK (2019), and confirmed by AGP for the current study, during operations, inflow rates will be a function of pit-development shape, volume, and rate of excavation, as well as hydraulic conductivity of the bedrock and the hydraulic gradient between the mine and surrounding surface water sources (lakes). Estimates of potential inflow rates to the pit were made at an "order of magnitude" accuracy level using analytical methods (Dupuit, 1863), and conceptual 2D and 3D groundwater flow simulations.

Hydraulic conductivity (K) values were derived from preliminary testwork. Given the location of the pit relative to the surrounding lakes, and the limited information on geology and structure currently available, a conservative approach was taken in estimating the pit-inflow. Inflow rates were estimated to reach a maximum of approximately 17,000 m^3/d. The potential for geological features to connect to the lakes was flagged as a risk to the Project in the form of unanticipated water management and environmental concerns.

Hydrogeological Evaluations

The following office-based data evaluation tasks are recommended to advance the Project to an FS level:

- Updating the existing 3D lithological and/or structural models to incorporate the results of all recent hydro-geotechnical drilling and/or an improved understanding of the deposit geology.
- Interpretation of structural and geotechnical data and development of a site geologic structural model incorporating major fault and shear structures.
- Consider transient numerical analyses, which take into account the actual mine sequencing.

16.5 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 First Mining personnel and the other contractors to select appropriate operating cost parameters for the Springpole open pit.

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 destinations. Process costs and a portion of the G&A costs were provided by SRK based on their benchmarking of other relevant studies and test results.

The parameters used are shown in Table 16-7. The revenue 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 226 t trucks using an approximate waste dump configuration to determine incremental hauls for mill feed and waste. The smelting terms and recovery assumptions are based on creating a gold and silver doré.

Table 16-7: Economic Pit Shell Parameters (US Dollars unless otherwise noted)

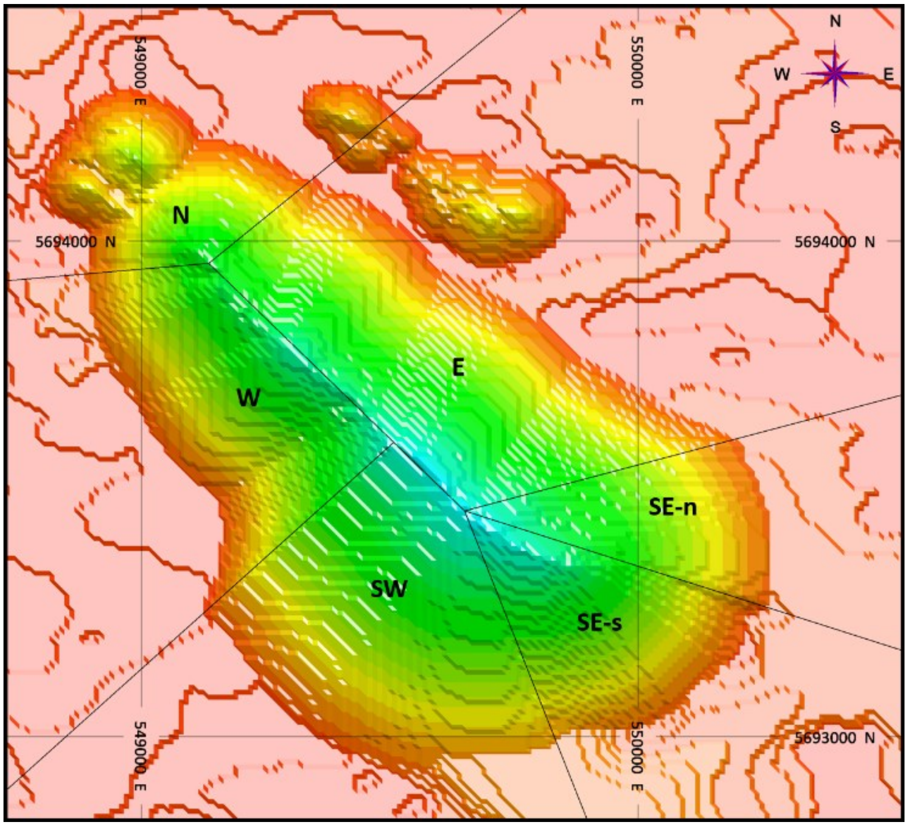
Description	Units	Value		
Exchange rates				
CDN	USD$ =	1.2987		
Resource Model				
Block classification used		M+I		
Block Model Height		6		
Mining Bench Height		12		
Metal Prices			**Gold**	**Silver**
Price	$/oz		1350.00	20.00
Royalty	%		3%	3%
Smelting, Refining, Transportation				
Payable	%		99.5%	98.0%
Minimum Deduction	unit, g/dmt		0	0
Transportation and Refining	$/oz		5.00	0.00
Net Price Calculation				
Payability	%		99.5%	98.0%
Precious Metal Deduction	$/oz		6.75	0.40
Transportation and Refining	$/oz		5.00	0.00
Subtotal Price	$/oz FOB Mine		1338.25	19.60
less Royalty	$/oz FOB Mine		40.15	0.59
Net Price	$/oz FOB Mine		1298.10	19.01
	$/g FOB mine		41.73	0.61
	CDN$/g FOB mine		54.20	0.79
Metallurgical Information				
Recovery	%		88.0%	93.0%
Power Cost				
Cost of power	CDN$/Kwhr	0.08		
Fuel Cost				
Diesel Fuel Cost to site	CDN$/ l	0.80		
Mining Cost *				
Waste Base Rate - 400m Elevation	CDN$/t	1.67		
Incremental Rate - above	CDN$/t/6m bench	-		
Incremental Rate - below	CDN$/t/6m bench	0.0199		
Mill Base Rate - 400m Elevation	CDN$/t	1.62		
Incremental Rate - above	CDN$/t/6m bench	-		
Incremental Rate - below	CDN$/t/6m bench	0.0202		
Processing **				
Processing Cost	CDN$/t ore	15.38		
General and Administrative Cost				
G&A Cost	CDN$/t ore	1.00		
Total Process and G&A				
Process + G&A	CDN$/t ore	16.39		

* mining costs based on using 226 t haul trucks.

** process costs based on 30 ktpd dry throughput.

Wall slopes for pit optimization were based on the assessment described in Section 16-3. Allowances were made for ramps in the slopes to determine an overall angle for use in the LG routine. The slope domain locations are shown in Figure 16-8 while the overall slope angle calculations are shown in Table 16-8.

Figure 16-8: Location of Slope Domains



Source: AGP, 2021

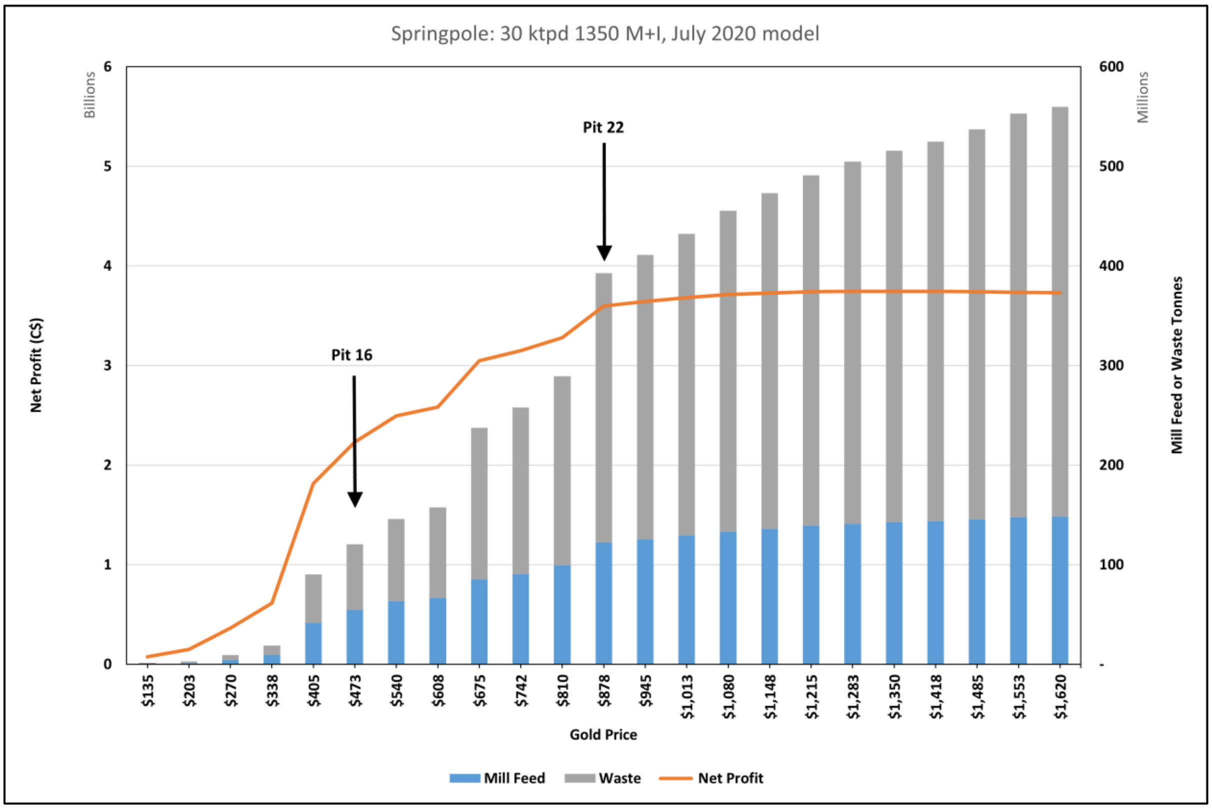
Table 16-8: Overall Slopes for Economic Pit Shells

Design Sector	Zone Code	Inter-Ramp Angle (degrees)	Haul Roads In Slope	Slope Height (m)	Overall Slope (degrees)	Rock Mass Rating (estimate)
OB	1	30.0	0	12	30	
N	2	54.2	1	300	50	good
E	3	51.0	2	375	45	fair
SE-n	4	38.9	2	375	35	poor to extremely poor
SE-s	5	44.9	2	375	40	fair to poor
SW	6	38.9	2	375	35	extremely poor to poor
W	7	51.6	2	340	45	fair to poor

Note: 35.4 m haul road width

Nested LG pit shells were generated to examine sensitivity to the gold and silver prices with base case prices of USD$1,350/oz Au and USD$20.00/oz Ag. This was to gain an understanding of the deposit and highlight potential opportunities in the design process to follow. Indicated Mineral Resources that were undiluted were used in the analysis. The net prices were 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 USD$135/oz up to USD$1,620/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 USD$1,350/oz Au and USD$20.00/oz Ag. Mill feed/waste tonnages and net profit were plotted against gold price and are displayed in Figure 16-9.

Figure 16-9: Springpole Profit vs Price by Pit Shell



Source: AGP, 2021

Figure 16-12 illustrates various break points in the pit shells. With each incremental pit shell, the waste tonnage, mill tonnage, and undiscounted net profit also increased up to the base price of USD$1,350/oz Au. In the case of the first break point shown at USD$473/oz Au, the cumulative waste tonnage is 65.9 Mt, with a corresponding mill feed tonnage of 54.7 Mt or a strip ratio of 1.2:1. The net profit also increased beyond this point showing that there was still value to be obtained by going with a higher metal price or an additional phase. This break point represented 60% of the net value of a

$1,350/oz Au pit but with only 18% of the waste of the larger pit shell. This pit shell was used to guide the design of the first phase of the main pit.

The next selected break point was at USD$878/oz Au. The incremental waste tonnage from the first break point is 204.7 Mt, with a corresponding increase in mill feed tonnage of 67.5 Mt or a strip ratio of 3.0:1. The cumulative net value of the first two break points was 96% of the USD$1,350/oz Au pit shell but with only 73% of the waste movement of the larger pit required. This pit shell was used for the pit design of the second /ultimate phase in the main pit. The incremental net value is not considered significant for the next higher pit prices, particularly when discounting is accounted for.

16.6 Dilution

The open pit resource model was provided as an undiluted percentage type model. 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 6 m high. Mining would be completed on 12 m lifts for waste and 6 m lifts for mill feed 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 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 a waste block, dilution would rise to 20%. Three or four sides in contact with a waste block would result in 30% and 40% dilution, respectively. Four sides represent an isolated block of mill feed.

All mineralized blocks 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 gold and silver 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 CDN$0.01/t cut-off represents the marginal cut-off grade to flag initial feed and waste blocks.

AGP applied a three-pass approach to define all block diluted grades and percentages. The three-pass dilution calculations are summarized as follows:

- For the first pass of dilution calculations, the marginal cut-off grade was used to code a FLAG model item, where (1) represents a mill feed block, (2) a waste block within mineralized material, or (3) a default waste block outside of mineralization.
- For the second pass of dilution calculations, two contact counters were defined. The count of waste blocks touching each ore block was stored in item DILBK, while the count of ore blocks touching each waste block was stored in item DILBO.

- For the third pass of dilution calculations, new diluted items were defined and stored in DORE%, DWAS%, DAU, and DAG. The block FLAG values were used to determine dilution as flows:
  - if FLAG = 1 (mill feed block), the number of waste blocks in contact was used to determine a dilution percentage with no grade to add 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.
  - if FLAG = 2 (waste block within mineralization), the number of feed blocks in contact was used to determine a dilution percentage at the waste block grade. The assumption was that only the feed contact sides of the mineralized waste block would be added as dilution.
  - if FLAG = 3 (waste block), a default block value would be defined as DORE%=0 and diluted gold (DAU) and silver (DAG) grades of 0 g/t.

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 pits, the diluted feed contained 4.9% more tonnes and 3.8% lower gold grade than the in-situ feed summary. The grade dilution percentage is lower than the feed tonnage percentage since the mineralized waste blocks included some grade. AGP considers these dilution percentages to be reasonable considering the nature of the mineralization.

16.7 Pit Designs

Pit designs were developed for the main pit as well as the small satellite pit immediately to the northeast using a 12m bench height. The initial pit phase design included only the small satellite pit. This small pit is also above the Springpole Lake level so is convenient from an early mining perspective. The main pit has been divided into phases 2 and 3, with phase 3 being the ultimate pit. The pit optimization shells used to guide the ultimate pit were also used to outline areas of higher value for targeted early mining and phase development.

Geotechnical parameters discussed in Section 16.3.5 were applied to pit designs as shown in Table 16-9. It should be noted that the lowest RQD values were observed in the SE-n and SW design sectors, which is why these slopes are shallower than the other design sectors.

Table 16-9: Pit Design Slope Criteria

Design Sector	Zone Code	Inter-Ramp Angle (degrees)	Bench Face Angle (degrees)	Height Between Berms (m)	Catch Bench Width (m)
OB	1	30.0	55	12	12.38
N	2	54.2	70	24	8.57
E	3	51.0	70	24	10.73
SE-n	4	38.9	70	24	21.01
SE-s	5	44.9	70	24	15.33
SW	6	38.9	70	24	21.01
W	7	51.6	70	24	10.27

Note: 12m bench heights during mining.

Equipment sizing for ramps and working benches is based on the use of 226 t rigid frame haul trucks. The operating width used for the truck is 8.3 m. This means that single lane access is 27.1 m (2x operating width plus berm and ditch) and double lane widths are 35.4 m (3x operating width plus berm and ditch). Ramp gradients are 10% in the pit and dump for uphill gradients. Working benches were designed for 35 to 40 m minimum mining width on pushbacks.

Tonnes and grade for the designed pit phases are reported in Table 16-10 using the diluted tonnes and grade from the resource model. Positive marginal block values from the optimization run were used to delineate mill feed from waste.

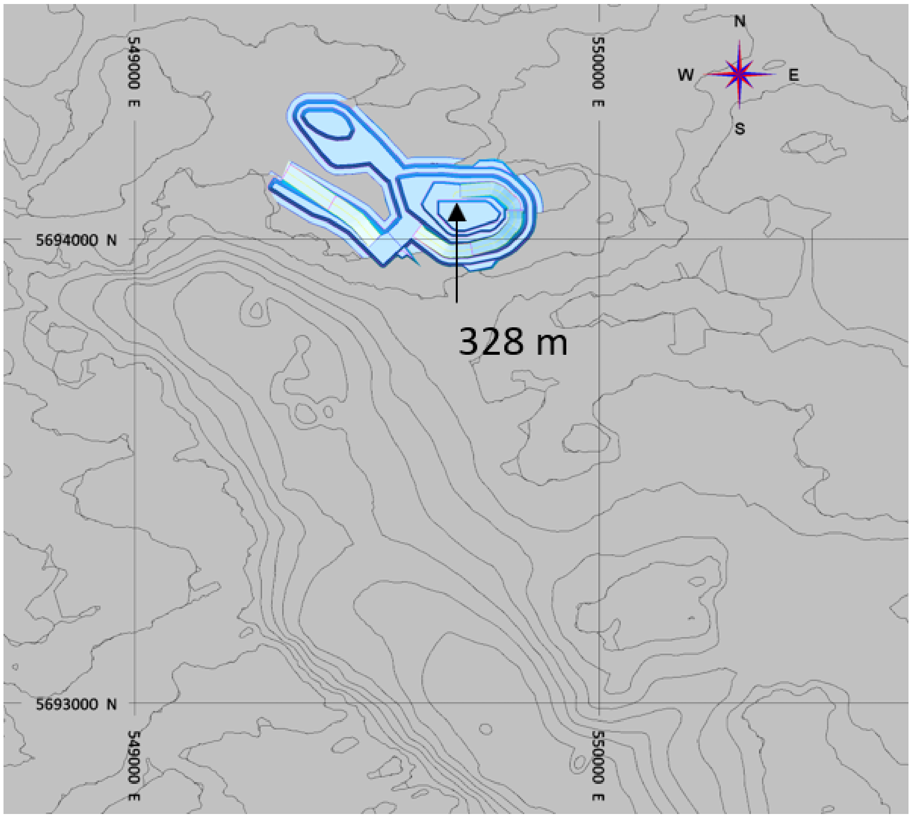
Table 16-10: Pit Phase Tonnage and Grades

Phase	Mill Feed (Mt)	Au (g/t)	Ag (g/t)	Waste (Mt)	Total (Mt)	Strip Ratio
1	2.1	1.19	0.97	6.6	8.7	3.10
2	64.7	1.05	5.04	103.1	167.9	1.59
3	54.8	0.87	5.62	165.7	220.4	3.02
Total	**121.6**	**0.97**	**5.23**	**275.4**	**397.0**	**2.26**

16.7.1 Phase 1

Phase 1 is the first phase mined in the schedule and comprises the small satellite pit to the northeast. The phase will be mined down to the 328 masl elevation. All waste and mill feed access will be on the west side of the pit in a slot configuration. The slot is the start of the ramp system for the entire pit design (refer to Figure 16-10).

Figure 16-10: Phase 1 Design

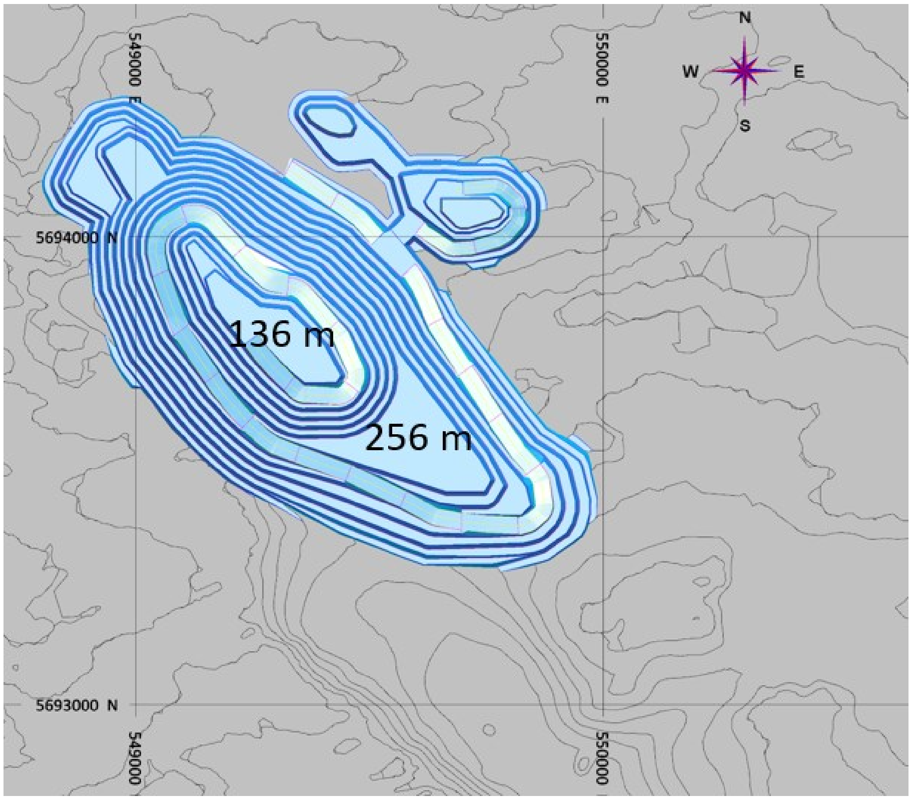


Source: AGP, 2021

16.7.2 Phase 2

Phase 2 is the second phase mined in the schedule and targets the upper and northern portion of the deposit. This phase will be mined down to the 136 masl elevation. Mine access is provided by the main ramp which is a continuation of the slot portion of the prior phase ramp. The ramp continues from 376 masl elevation on the southeast side of phase 2, clockwise to the bottom of the phase (refer to Figure 16-11).

Figure 16-11: Phase 2 Design

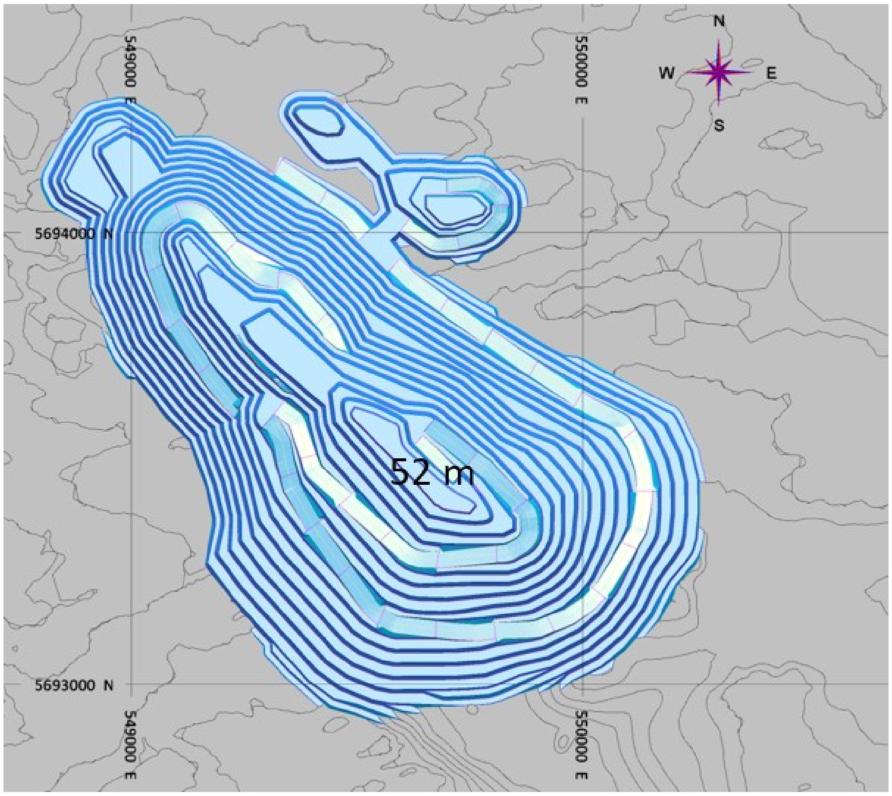


Source: AGP, 2021

16.7.3 Phase 3

Phase 3 is the third and final phase mined in the schedule. This phase expands the pit to southwest targeting the lower portion of the deposit. Phase 3 is mined down to the 52 masl elevation. Access to this phase is provide by a ramp that branches off the prior ramp on the southeast side of the phase from the 376 masl elevation (refer to Figure 16-12).

Figure 16-12: Phase 3 Design



Source: AGP, 2021

16.8 Waste Rock Storage Facility Design

Waste rock and filtered tails will be stored in the WSF that will be located adjacent to the pit to the northwest. A small portion of NAG waste rock will also be stored within the northern portion of the pit after Phase 2 has been depleted. Most of the NAG material will be used to construct embankments of the WSF using interior slope of 3H:V1 and exterior slope of 2H:1V. PAG waste rock and filter tails will be placed within the embankments. A swell factor of 22% (considering some compaction) is used for waste rock and overburden material to estimate volume requirements. A density of 1.6 t / m^3 is used for filtered tailings.

The in-pit waste storage will have a capacity of 9.8 Mm^3 NAG material late in the mining schedule. It will be built to a maximum elevation of 388 masl to conserve eventual lake habitat. The proportion of NAG versus PAG in the mine waste is shown in Table 16-11.

Table 16-11: Waste Characterization

Waste Material	Tonnage (Mt)
NAG Overburden	10.0
NAG Rock	164.9
PAG Overburden	10.9
PAG Rock	89.6
Total	**275.4**

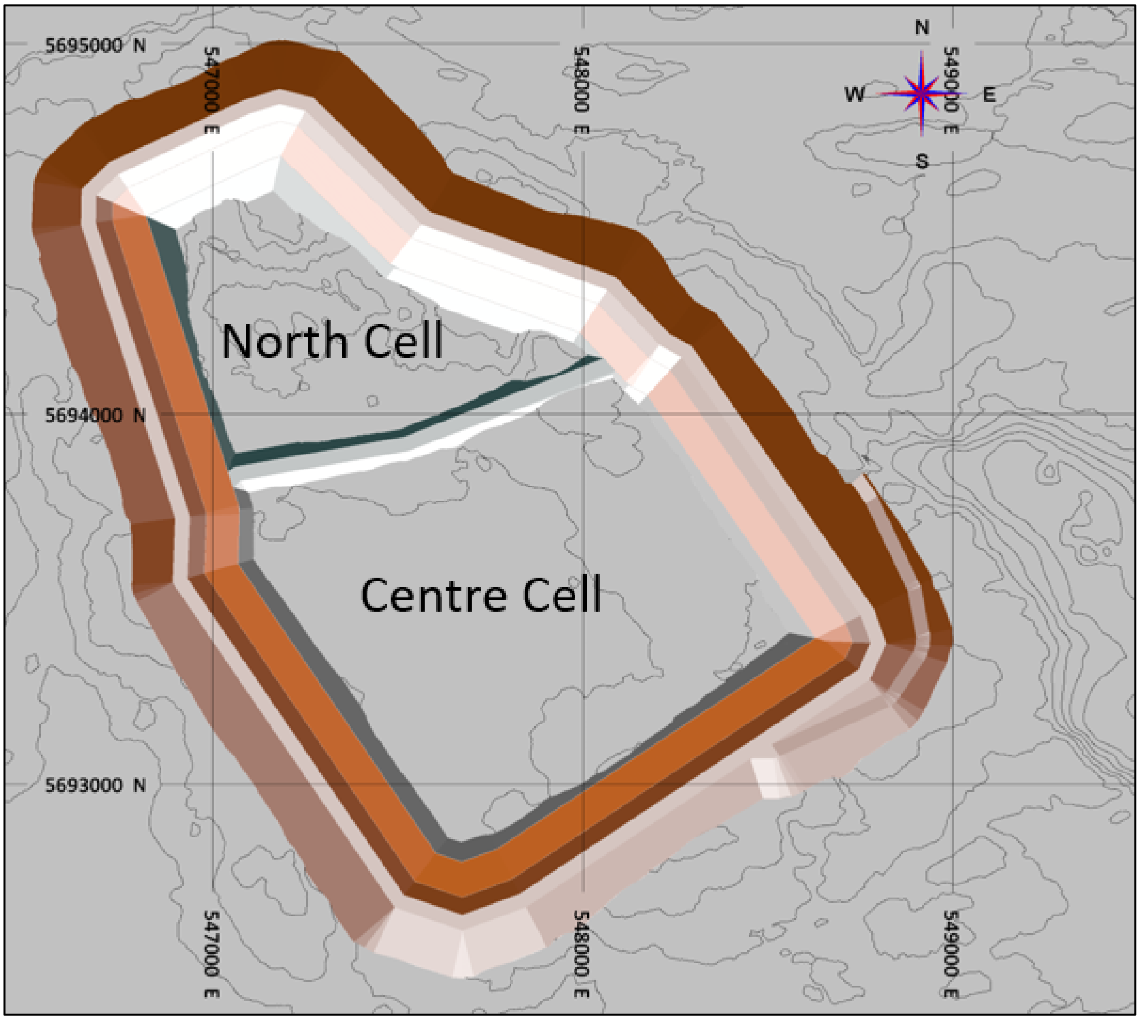
The WSF is designed to be constructed in three phases. The first phase is the central cell which will be built to the 425 masl elevation using 4.3 Mm^3 of NAG material and will hold 14.1 Mm^3 of PAG material. The second phase will be the north cell which will be built to 425 masl elevation using 3.6 Mm^3 of NAG material and will store 5.3 Mm^3 of PAG material. The third phase will cover the first two phases to an elevation of 470 masl using 37.7 Mm^3 of NAG material and will store 98.5 Mm^3 of PAG material.

The third phase embankment will be constructed in two "rings" so that inner ring can be advanced to the required elevations throughout the production schedule while outer ring follows. Access to the structure will be provided by a ramp located on the southeast corner of the WSF. Refer to Table 16-12, Figure 16-13, Figure 16-14, Figure 16-15, and Figure 16-16.

Table 16-12: Waste Storage Capacity

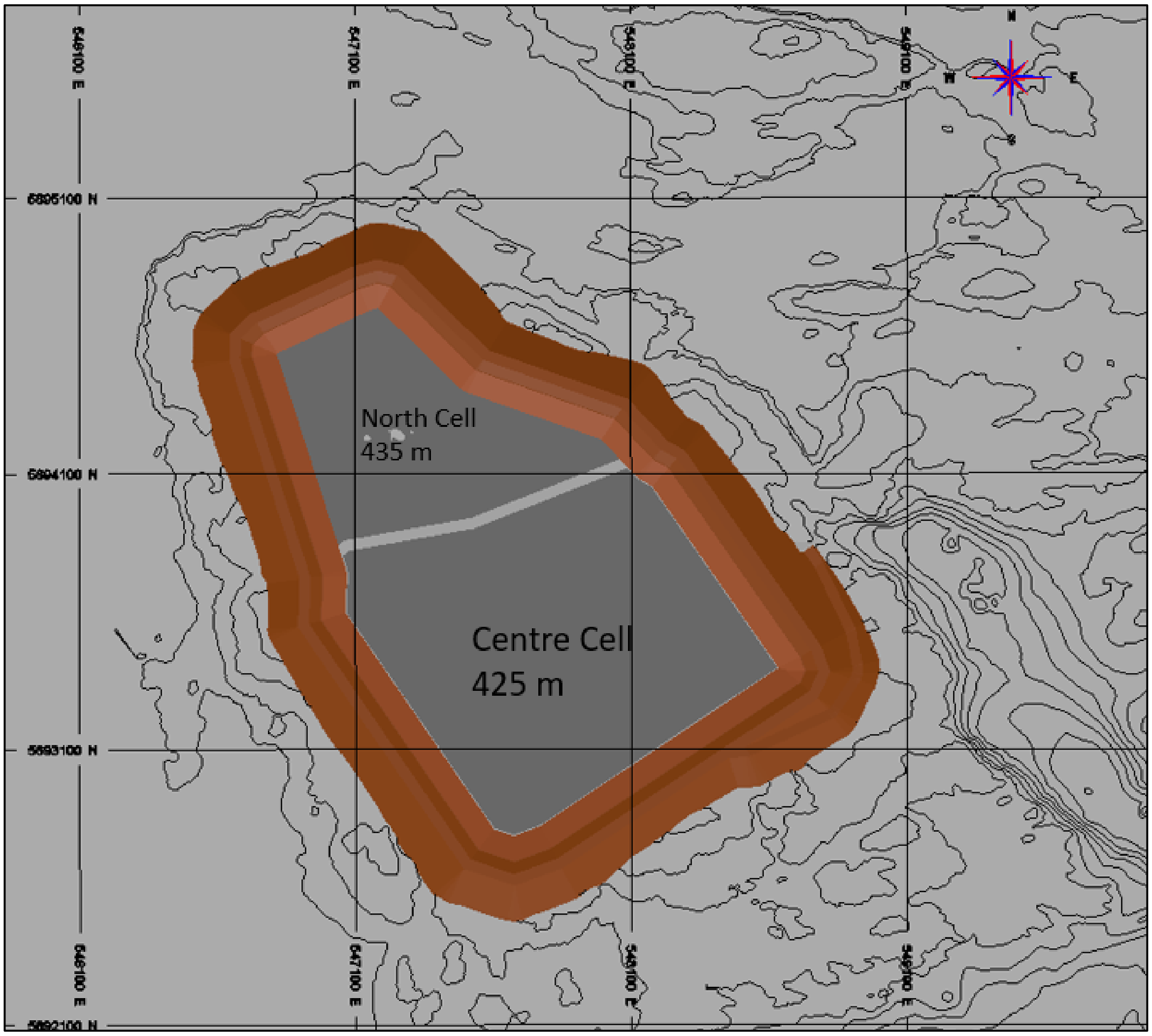
Facility	Units	WD01	North Pit Backfill
Waste Storage Capacity	Mm^3	196.6	9.8
Maximum Elevation	masl	470	388

Figure 16-13: WSF Embankment Phasing



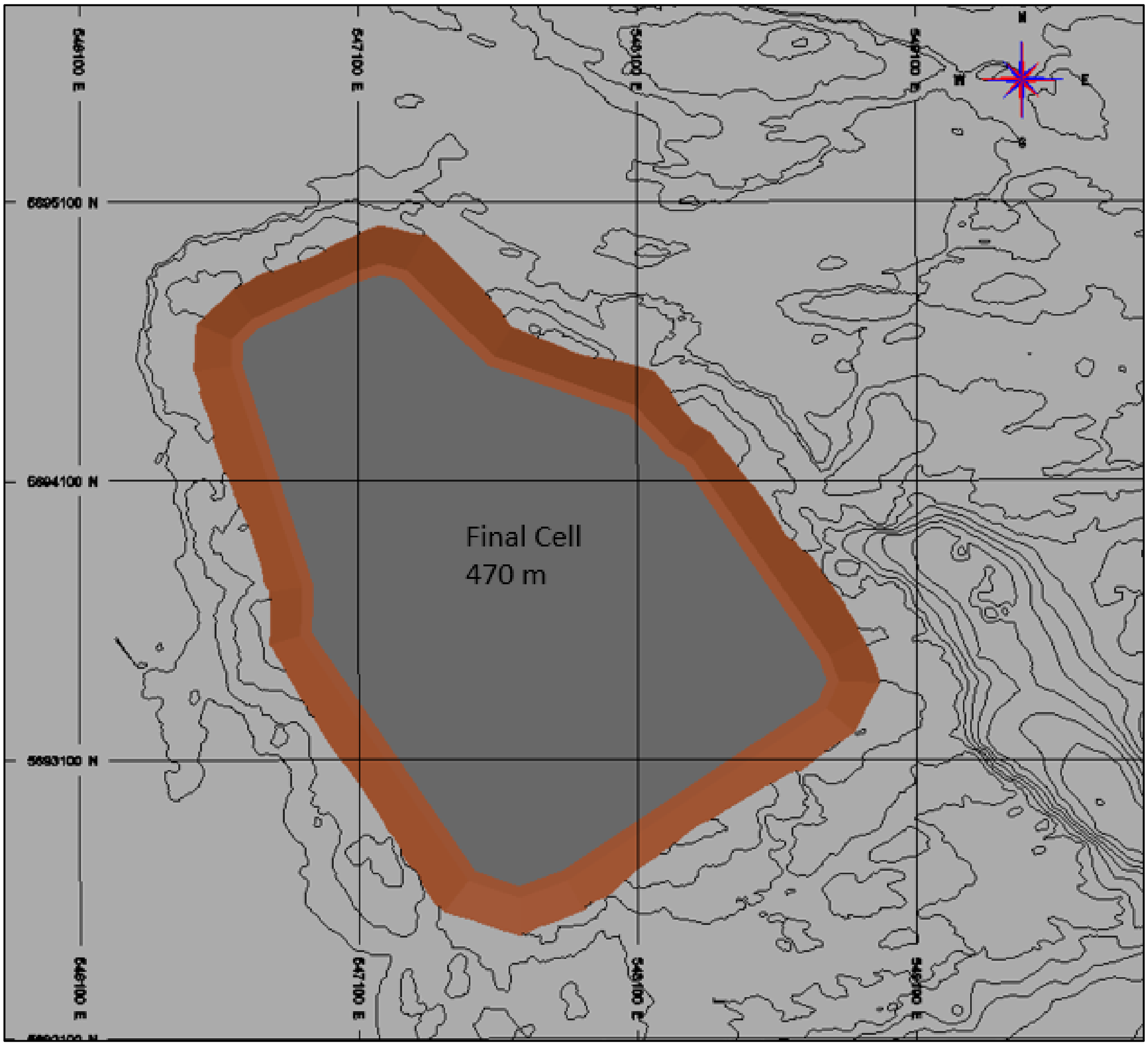
Source: AGP, 2021

Figure 16-14: WSF Content Configuration for North and Centre Cells



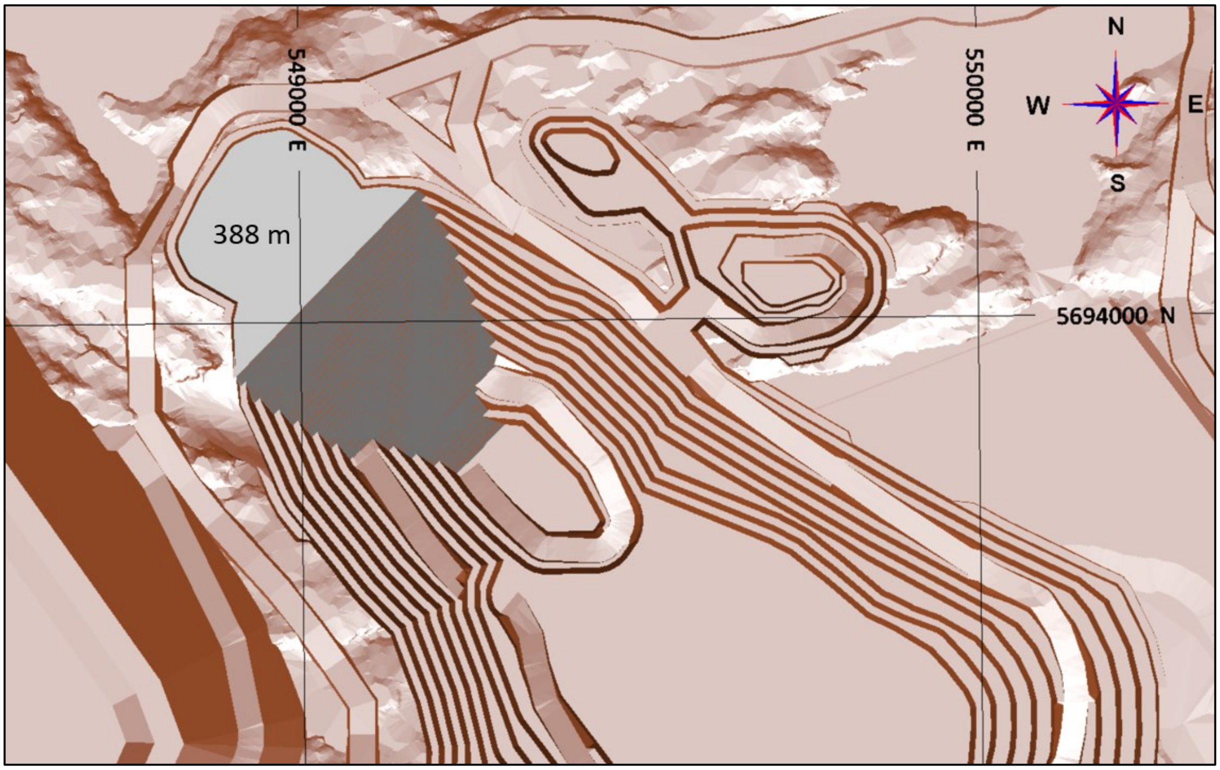
Source: AGP, 2021

Figure 16-15: WSF Content Configuration for Final Cell



Source: AGP, 2021

Figure 16-16: In-pit Rock Storage Facility



Source: AGP, 2021

16.9 Mine Schedule

The mine production schedule consists of 121.6 Mt of mill feed grading 0.97 g/t Au and 5.2 g/t Ag of the 12 years of mine life. The processing rate will be 10.95 Mt per year. Mine overburden and rock waste tonnage totals 275 Mt and will be placed in the WSF, and a small in-pit facility. The overall pit strip ratio will be 2.26:1.

The mining production schedule includes one year of pre-stripping, nine years of mining, and three years of processing stockpiled material. The plant feed will consist of material coming from the three pit phases and three stockpiles. In year 10, tonnage in the mine will be exhausted with processing material only coming from the stockpiles until the end of year 12.

Mill feed will be stockpiled during the pre-production year and throughout the production schedule as required. Three stockpiles will be located north of the plant site - high grade, medium grade, and low grade.

The stockpile diluted gold cut-off grades will be:

- High Grade Stockpile >= 0.8 g/t
- Medium Grade Stockpile < 0.8 g/t and >= 0.6 g/t

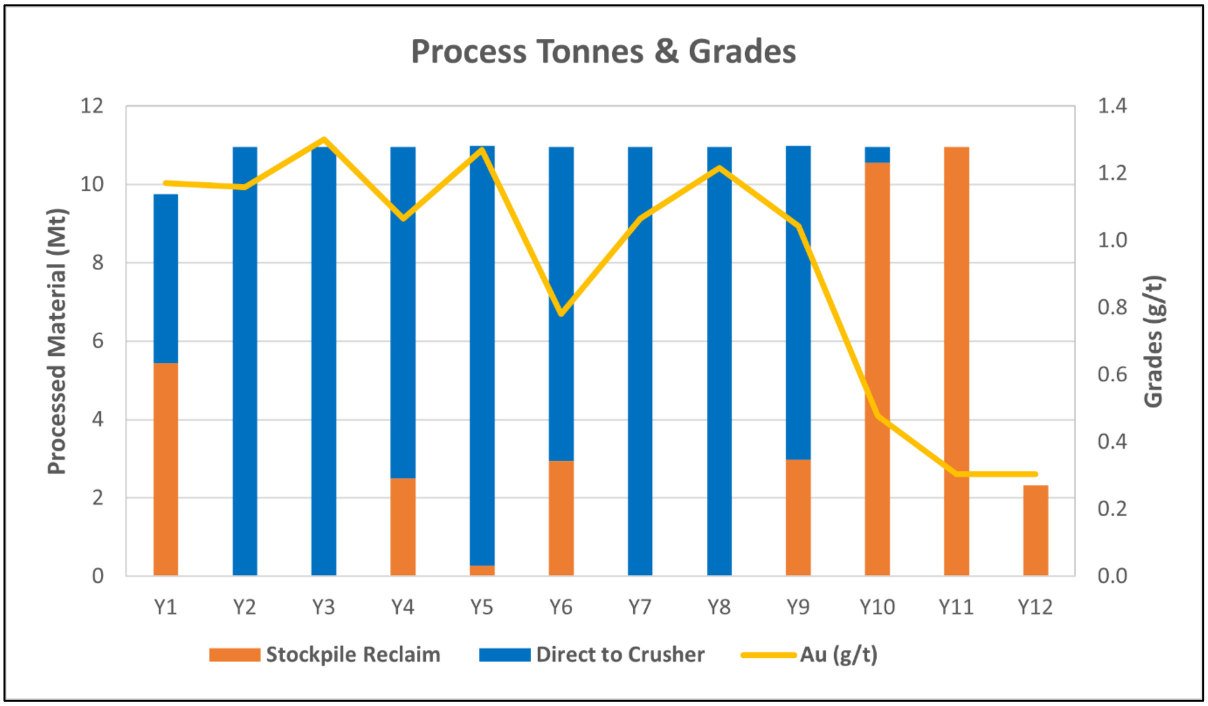
- Low Grade Stockpile < 0.6 g/t and >= 0.4 g/t

A peak stockpile capacity of 26.0 Mt will be reached near the end of year 8 (refer to Table 16-13 and Figure 16-17).

Table 16-13: Mine Schedule

		Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Total
Mining Summary	**Waste (Mt)**	**15.8**	**31.6**	**34.2**	**33.6**	**39.9**	**39.0**	**41.9**	**31.0**	**7.3**	**1.2**	**0.01**	**0.00**	**0.00**	**275**
	Mill Feed (Mt)	**4.54**	**7.98**	**15.82**	**16.42**	**10.09**	**11.18**	**8.07**	**19.04**	**19.31**	**8.77**	**0.39**	**0.00**	**0.00**	**121.6**
	Au (g/t)	1.01	0.96	0.92	1.00	1.08	1.25	0.89	0.79	0.98	0.97	0.83	0.00	0.00	**0.97**
	Ag (g/t)	1.1	3.8	4.6	4.6	6.2	6.5	5.1	4.5	6.8	6.4	4.8	0.0	0.0	**5.2**
	Total (Mt)	**20.3**	**39.6**	**50.0**	**50.0**	**50.0**	**50.1**	**50.0**	**50.0**	**26.6**	**10.0**	**0.40**	**0.00**	**0.00**	**397**
Processed Material	**Mill Feed (Mt)**	**0.00**	**9.75**	**10.95**	**10.95**	**10.95**	**10.98**	**10.95**	**10.95**	**10.95**	**10.98**	**10.95**	**10.95**	**2.32**	**121.6**
	Au (g/t)	0.00	1.17	1.16	1.30	1.06	1.27	0.78	1.07	1.22	1.04	0.48	0.30	0.30	**0.97**
	Ag (g/t)	0.0	3.2	5.7	5.6	5.6	6.6	4.4	5.4	8.3	6.6	3.7	2.8	2.8	**5.2**
Stockpile Balance	**Low Grade (Mt)**	**0.86**	**2.32**	**4.99**	**7.75**	**9.34**	**9.82**	**9.58**	**12.72**	**14.89**	**14.92**	**13**	**2**	**0**	
	Au (g/t)	0.27	0.27	0.29	0.29	0.29	0.29	0.29	0.30	0.30	0.30	0.30	0.30	0.30	
	Ag (g/t)	0.7	1.8	2.1	2.3	2.6	2.5	2.5	2.6	2.8	2.8	2.8	2.8	2.8	
	Medium Grade (Mt)	**1.58**	**0.46**	**2.66**	**5.37**	**2.92**	**2.64**	**0.00**	**4.95**	**8.17**	**8.91**	**0**	**0**	**0**	
	Au (g/t)	0.58	0.52	0.50	0.49	0.49	0.49	0.00	0.49	0.49	0.49	0.00	0.00	0.00	
	Ag (g/t)	1.1	1.3	1.8	2.1	2.1	2.1	0.0	3.7	3.8	3.8	0.0	0.0	0.0	
	High Grade (Mt)	**2.11**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**2.97**	**0.00**	**0**	**0**	**0**	
	Au (g/t)	1.64	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1.12	0.00	0.00	0.00	0.00	
	Ag (g/t)	1.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0	6.6	0.0	0.0	0.0	0.0	
Total Stockpile Reclaim	**(Mt)**	**0.00**	**5.44**	**0.00**	**0.00**	**2.50**	**0.27**	**2.95**	**0.00**	**0.00**	**2.97**	**10.56**	**10.95**	**2.32**	**37.96**
Total Material Movement	**(Mt)**	**20.32**	**45.00**	**50.00**	**50.00**	**52.50**	**50.41**	**52.95**	**50.00**	**26.62**	**12.94**	**10.96**	**10.95**	**2.32**	**434.97**
Total Stockpile Balance	**(Mt)**	**4.5**	**2.8**	**7.7**	**13.1**	**12.3**	**12.5**	**9.6**	**17.7**	**26.0**	**23.8**	**13.27**	**2.32**	**0.00**	**0**

Figure 16-17: Process Tonnage and Gold Grade



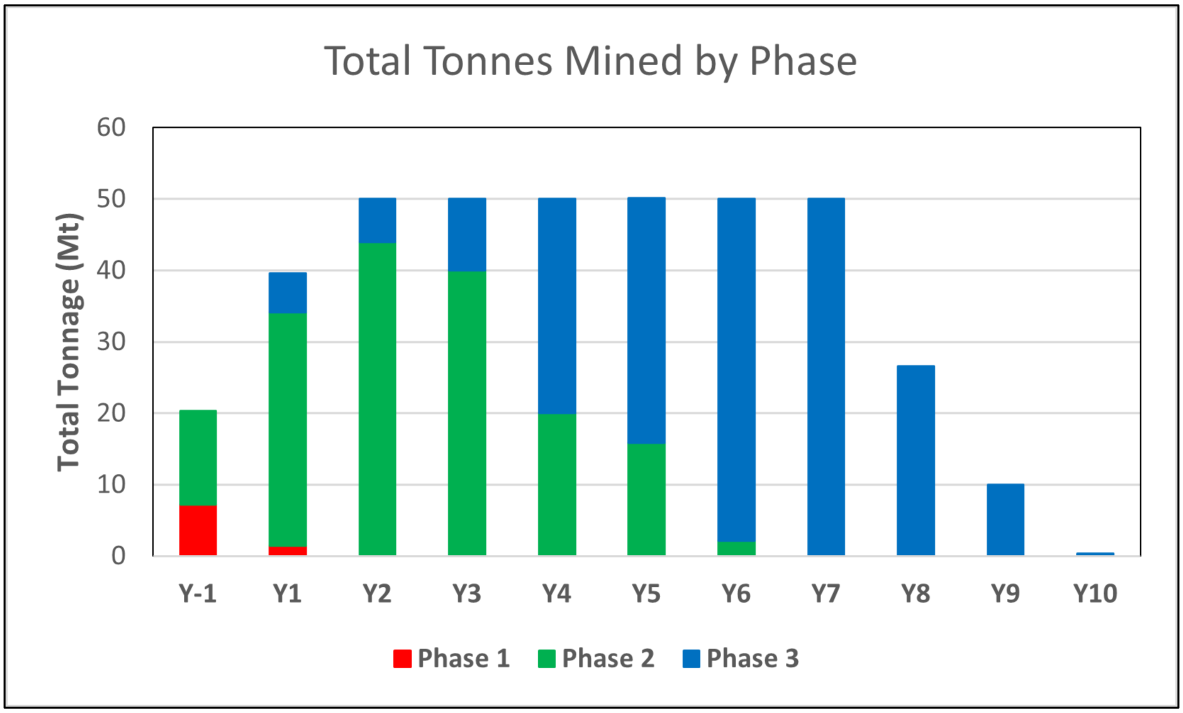
Source: AGP, 2021

The annual mining rate starts at 20.3 Mt/s in the pre-production year and reaches a peak of 50.1 Mt/a in year five. A maximum descent rate of six benches per year per phase is applied to ensure that reasonable mining operations and ore control will occur (refer to Table 16-14 and Figure 16-18).

Table 16-14: Total Tonnes Mined by Phase

Phase	Total Tonnage (Mt)											Total
	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	(Mt)
1	7.3	1.5										**8.7**
2	13.1	32.7	44.0	40.0	20.0	15.9	2.2					**167.9**
3		5.4	6.0	10.0	30.0	34.2	47.8	50.0	26.6	10.0	0.4	**220.4**
Total	**20.3**	**39.6**	**50.0**	**50.0**	**50.0**	**50.1**	**50.0**	**50.0**	**26.6**	**10.0**	**0.4**	**397.0**

Figure 16-18: Total Tonnes Mined by Phase



Source: AGP, 2021

Table 16-15 displays a summary of the reserve classification for the mill feed.

Table 16-15: Reserve Classification in Mill Feed

Reserve Class	Mill Feed (Mt)	Grade		Contained Ounces	
		Au (g/t)	Ag (g/t)	Au (Moz)	Ag (Moz)
Proven	0.0	0.00	0.00	0.00	0.0
Probable	121.6	0.97	5.23	3.80	20.5
Total	**121.6**	**0.97**	**5.23**	**3.80**	**20.5**

16.9.1 Quarry

An additional area has been included in the mine plan which is separate from the pit proper. This area is adjacent to the pit in the southeast corner. This area contains a large peat zone which will be used for later reclamation purposes.

The intent of mining in this area is to provide rock for various infrastructure projects initially. This includes:

- Haul roads to the:
  - pit
  - WSF
  - cofferdams

- fill material for the cofferdams
- fill material for construction purposes

This is mined in Year -3 and Year -2 of the overall schedule with the smaller equipment fleet.

The quarry will be reactivated in Year 10 with the larger mine fleet. The intention is to lower the area to a depth 3+ m below the proposed final lake level after mining is complete. Material in this final stage of excavation will be dumped into the pit to flatten the slopes of the pit before flooding at the end of the mine life.

Total material to be mined in the quarry is 12.1 Mt with 3.9 Mt mined in the pre-production period.

16.10 Mine Plan Sequence

The mine plan sequence will be as follows:

Year -3: Site construction is initiated which include access roads and cofferdams. Roads are built to provide access to the site, plant, camp pads, east side cofferdam, west side cofferdam, and an island road between the cofferdams. The WSF waste facility base area is cleared and grubbed. Turbidity curtains are installed in the lakes and lakes are de-fished. A borrow pit located on adjacent northeastern side of the future open pit provides 3.9 Mt for the construction. This tonnage has not been considered in the open pit tonnages shown previously.

Year -2: Site construction continues with overburden removal of the WSF/waste facility base area, base preparation cut and fill, and centre cell sidewall construction, and optional geomembrane installation. Base drilling and blasting provide 1.5 Mm^3 of material – 1 Mm^3 for base fill and 0.5 Mm^3 for side wall initiation of the centre cell. The de-fishing is completed, and the lakes are dewatered.

Year -1: Mining is initiated in phase 1 and 2 of the open pit. In this period, a total of 15.8 Mt of waste material is moved as the Project ramps up. As the processing plant is not yet operational, 4.54 Mt of material grading 1.01 g/t Au and 1.1 g/t Ag is stockpiled. Phase 1 and 2 of the open pit descend to the 364 masl and 376 masl elevations. NAG waste material is routed to construction of central and north embankment of the WSF waste facility and PAG material is placed in the centre cell. The central embankment is complete to the 425 masl elevation and the north embankment is nearly complete to the 422 masl elevation.

Year 1: Mining is initiated in phase 3 and continues in phase 2. Phase 1 is completed to the 328 masl elevation. Phase 2 and 3 descend to the 352 masl and 376 masl elevations, respectively. The processing plant is operational, and 9.75 Mt grading 1.17 g/t Au and 3.2 g/t Au are fed to it. A total of 31.6 Mt of waste is mined - NAG waste material is routed to the construction north and the final embankment. PAG waste and filter tails material is routed to centre cell and it reaches an elevation of 421 masl. The north embankment is complete to the 425 masl elevation and inner and outer phases of the final embankment raised to the 421 masl and 414 masl elevations, respectively.

Year 2: Mining continues in phases 2 and 3 and descend to the 316 masl and 376 masl elevations. Feed of 10.95 Mt grading 1.16 g/t Au and 5.7 g/t Ag is processed, and 34.2 Mt of waste is mined. NAG material has raised the final embankment to the 428 masl and 416 masl elevations. PAG waste and filter tails have filled the central and north cells to the 426 masl elevation. The final cell is initialized.

Year 3: Mining continues in phases 2 and 3 and descend to the 268 masl and 364 masl elevations. Feed of 10.95 Mt grading 1.30 g/t Au and 5.6 g/t Ag is processed, and 33.6 Mt of waste is mined. NAG material has raised the final embankment to the 435 masl and 420 masl elevations. PAG waste and filter tails have filled the final cell to 433 masl elevation.

Year 4: Mining continues in phases 2 and 3 and descend to the 220 masl and 328 masl elevations. Feed of 10.95 Mt grading 1.06 g/t Au and 5.6 g/t Ag is processed, and 39.9 Mt of waste is mined. Head grades are lower in year 4 as 2.5 Mt of material is taken from stockpiles. NAG material has raised the final embankment to the 442 masl and 433 masl elevations. PAG waste and filter tails have filled the final cell to 440 masl elevation.

Year 5: Mining continues in phases 2 and 3 and descend to the 160 masl and 304 masl elevations. Feed of 10.98 Mt grading 1.27 g/t Au and 6.6 g/t Ag is processed, and 39.0 Mt of waste is mined. NAG material has raised the final embankment to the 448 masl and 447 masl elevations. PAG waste and filter tails have filled the final cell to 446 masl elevation.

Year 6: Mining continues in phases 2 and 3 and descend to the 136 masl and 256 masl elevations. Phase 2 is exhausted. Feed of 10.95 Mt grading 0.78 g/t Au and 4.4 g/t Ag is processed, and 41.9 Mt of waste is mined. Head grades are lower in year 6 as 2.95 Mt of material is taken from stockpiles. NAG material has raised the final embankment to the 466 masl elevation as a single phase. PAG waste and filter tails have filled the final cell to 452 masl elevation.

Year 7: Mining proceeds in phase 3 descending to the 184 masl elevation. Feed of 10.95 Mt grading 1.07 g/t Au and 5.4 g/t Ag is processed, and 31.0 Mt of waste is mined. A small portion of the NAG material has raised the final embankment to the 470 masl elevation and completed the embankment. The remaining 8.1 Mm^3 of the NAG waste is stored in pit at the north end. PAG waste and filter tails have filled the final cell to 457 masl elevation.

Year 8: Mining continues in phase 3 descending to the 124 masl elevation. Feed of 10.95 Mt grading 1.22 g/t Au and 8.3 g/t Ag is processed, and 7.3 Mt of waste is mined. 1.6 Mm^3 of the NAG waste is stored in pit at the north end. PAG waste and filter tails have filled the final cell to 461 masl elevation.

Year 9: Mining continues in phase 3 descending to the 64 masl elevation. Feed of 10.98 Mt grading 1.04 g/t Au and 6.6 g/t Ag is processed, and 1.2 Mt of waste is mined. Head grades are trending lower in year 9 and onwards as stockpiles are being reclaimed. 57km^3 of the NAG waste is stored in-pit at the north end. PAG waste and filter tails have filled the final cell to 464 masl elevation.

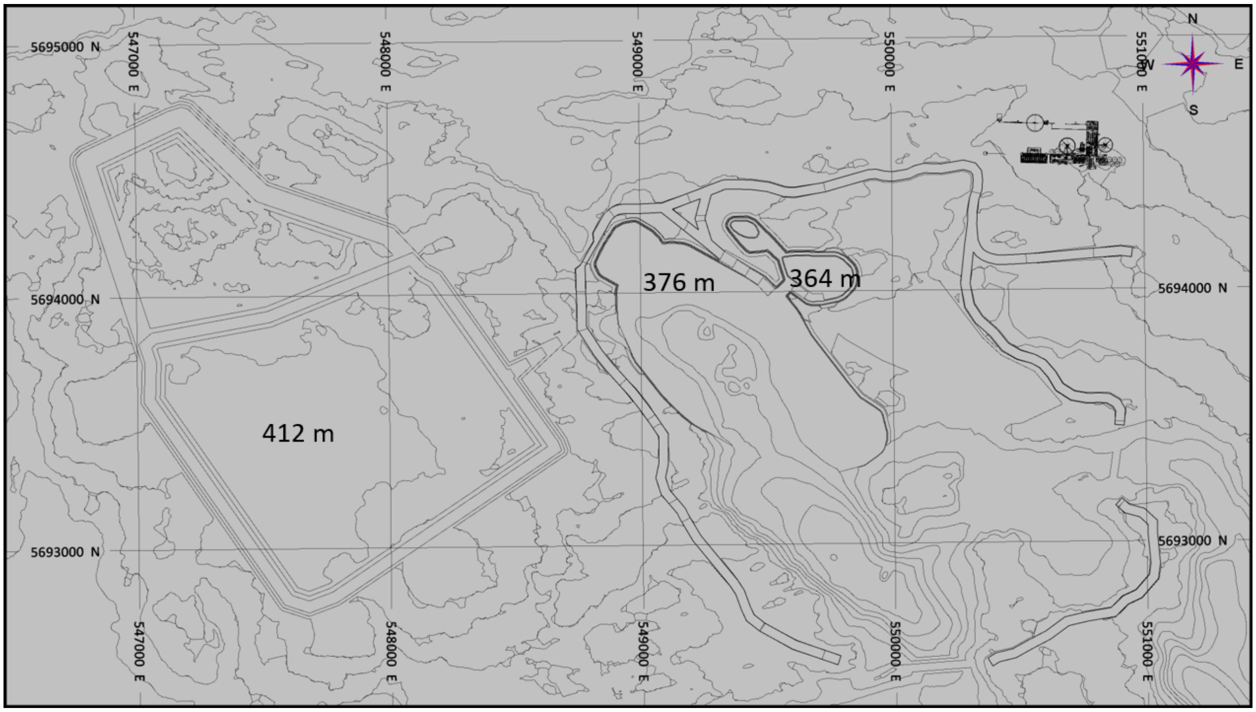
Year 10: Mining is complete in phase 3 descending to the 52 masl elevation. Feed of 10.95 Mt grading 0.48 g/t Au and 3.7 g/t Ag is processed, and 0.01 Mt of waste is mined. The majority of mill feed is from stockpile reclaim (10.56 Mt). PAG waste and filter tails have filled the final cell to 467 masl elevation.

Year 11: Processing continues solely from stockpiles. Feed of 10.95 Mt grading 0.30 g/t Au and 2.8 g/t Ag is processed. Filter tails have filled the final cell to 469 masl elevation. A section of land between the cofferdams and the southeast side of the ultimate pit is mined.

Year 12: Processing is complete from stockpiles. Feed of 2.32 Mt grading 0.30 g/t Au and 2.8 g/t Ag is processed. Filter tails have filled the final cell to the 470 masl design crest elevation.

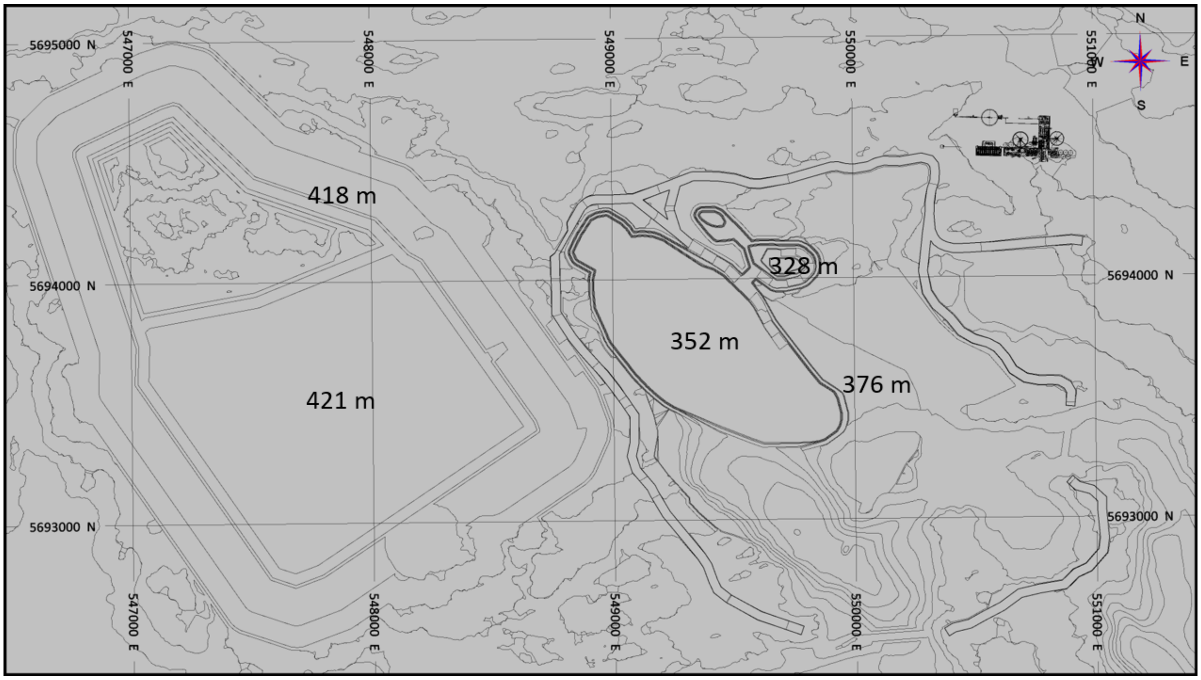
The annual periods are displayed from end of pre-production to the end of processing are displayed in Figures 16-19 to Figure 16-31.

Figure 16-19: End of Pre-Production



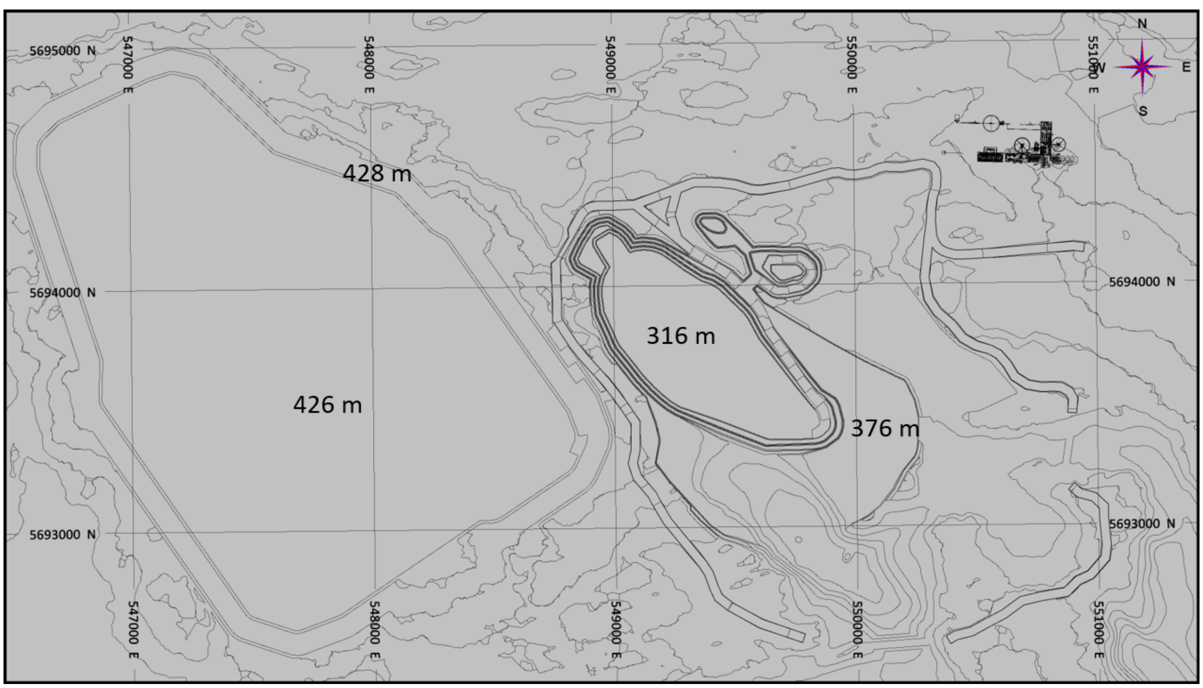
Source: AGP, 2021

Figure 16-20: End of Year 1



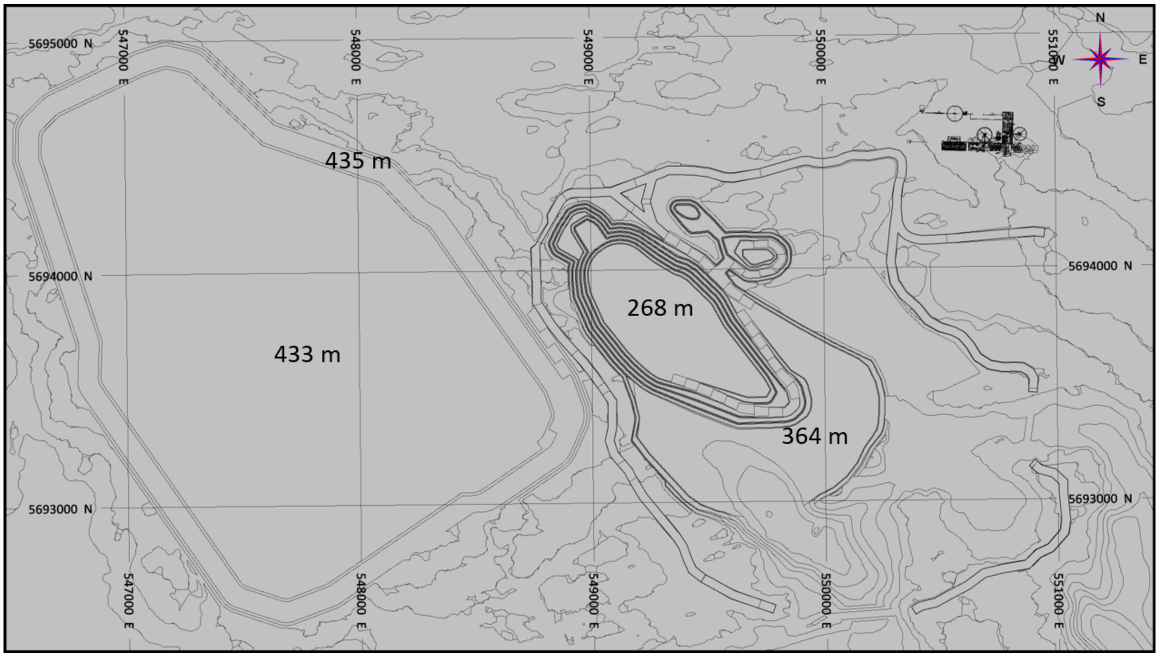
Source: AGP, 2021

Figure 16-21: End of Year 2



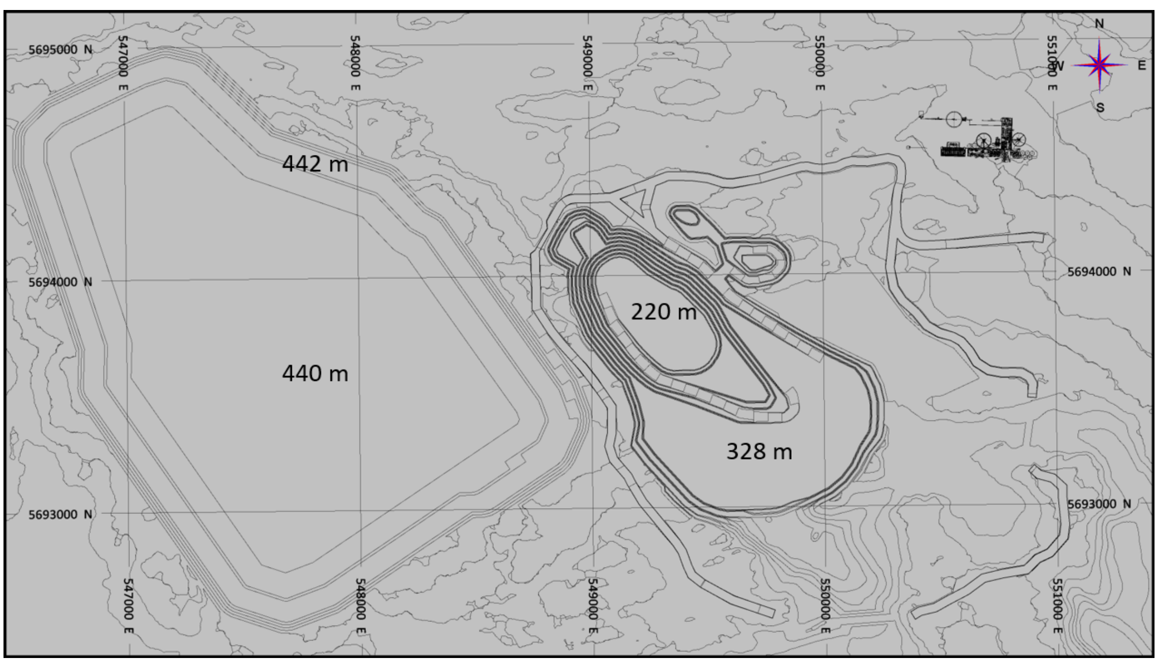
Source: AGP, 2021

Figure 16-22: End of Year 3



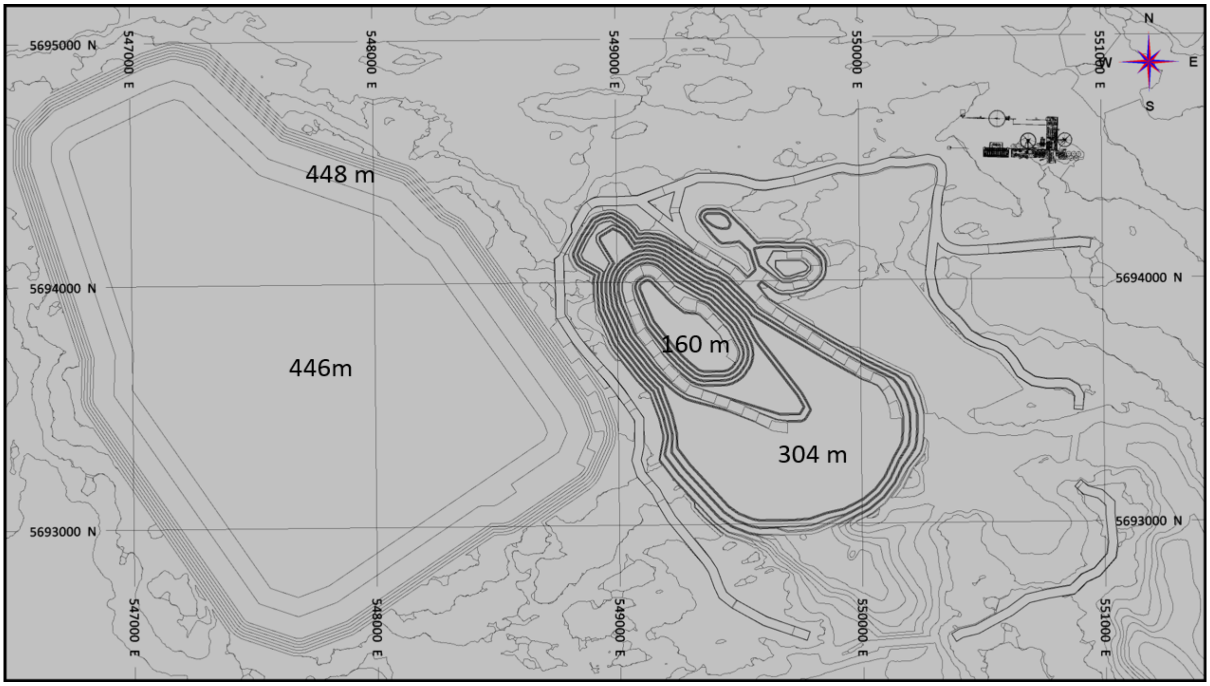
Source: AGP, 2021

Figure 16-23: End of Year 4



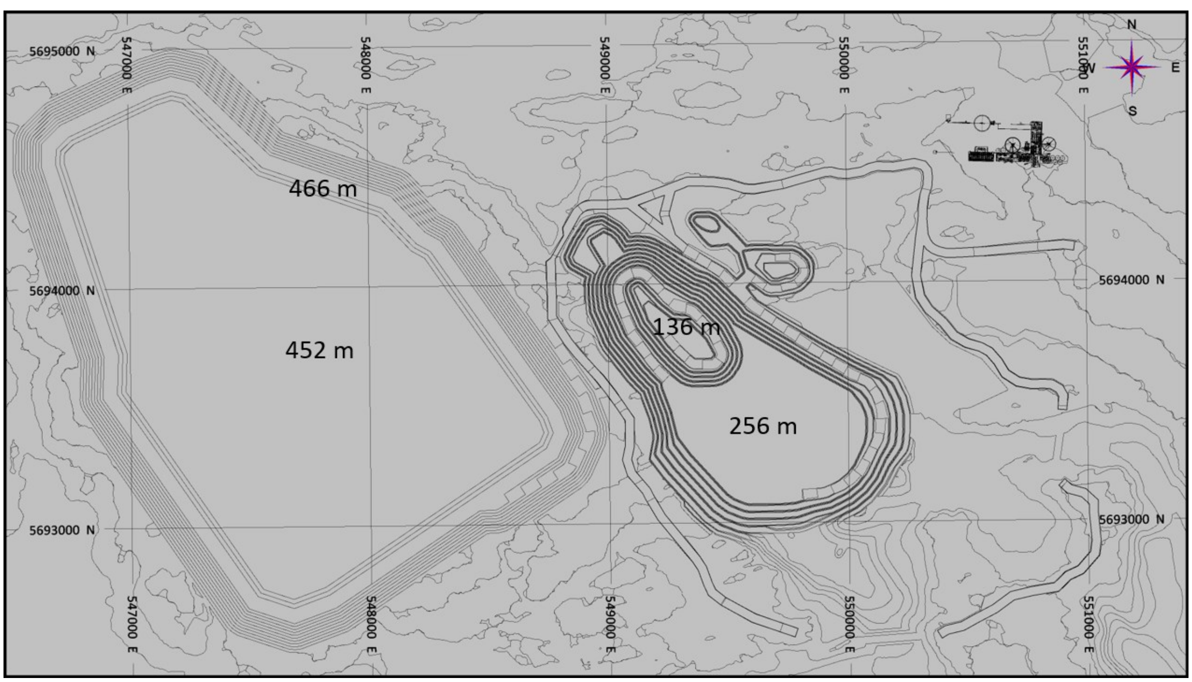
Source: AGP, 2021

Figure 16-24: End of Year 5



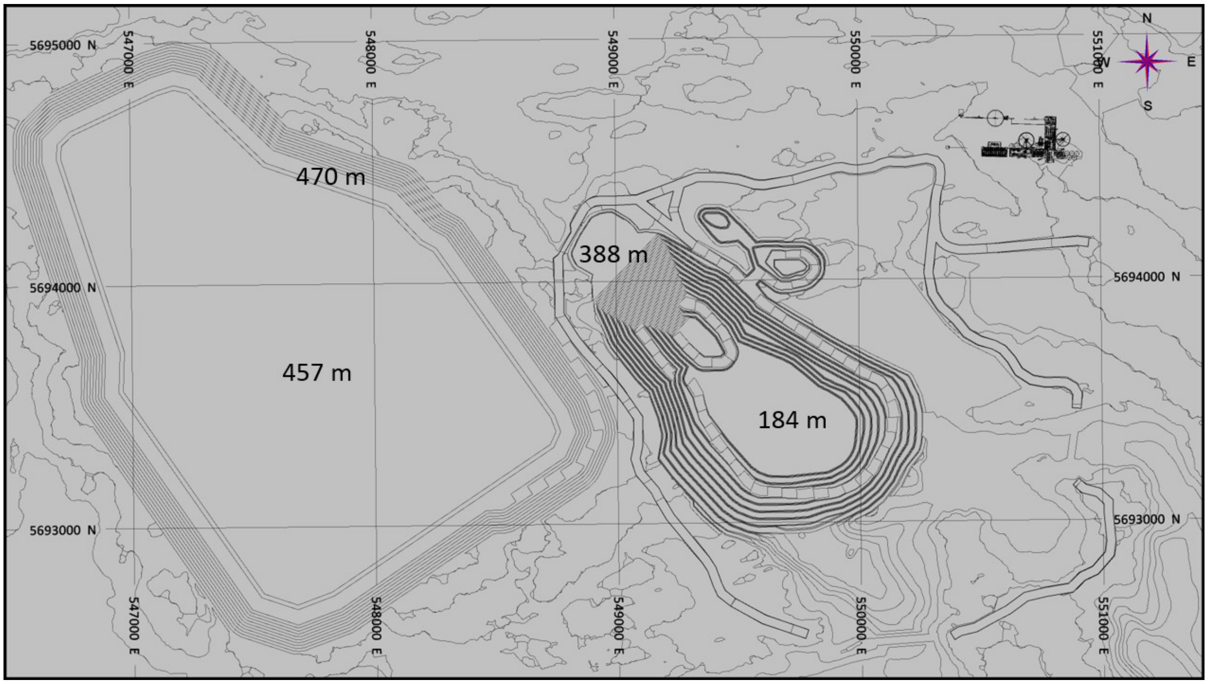
Source: AGP, 2021

Figure 16-25: End of Year 6



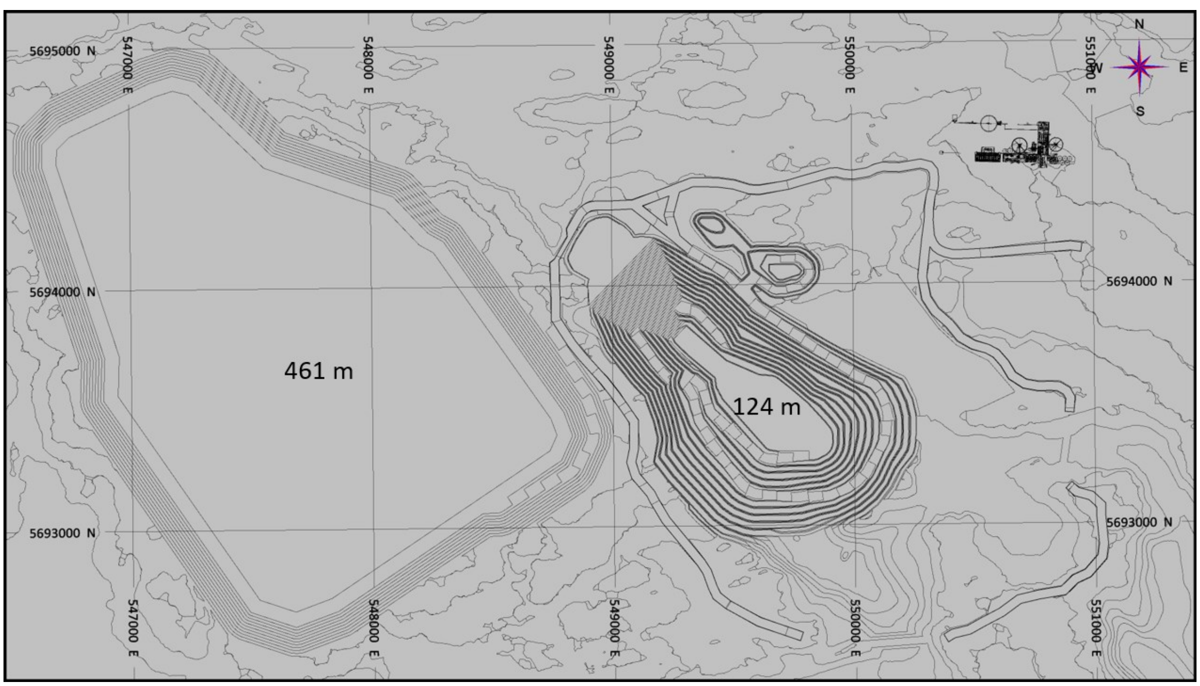
Source: AGP, 2021

Figure 16-26: End of Year 7



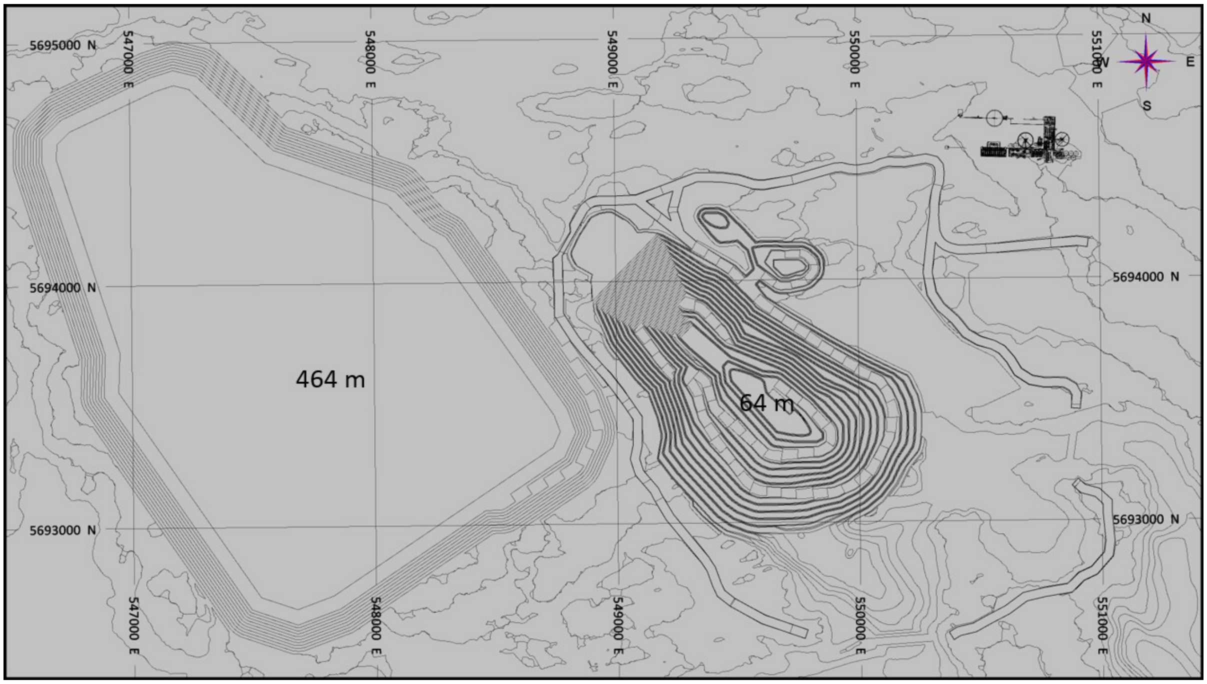
Source: AGP, 2021

Figure 16-27: End of Year 8



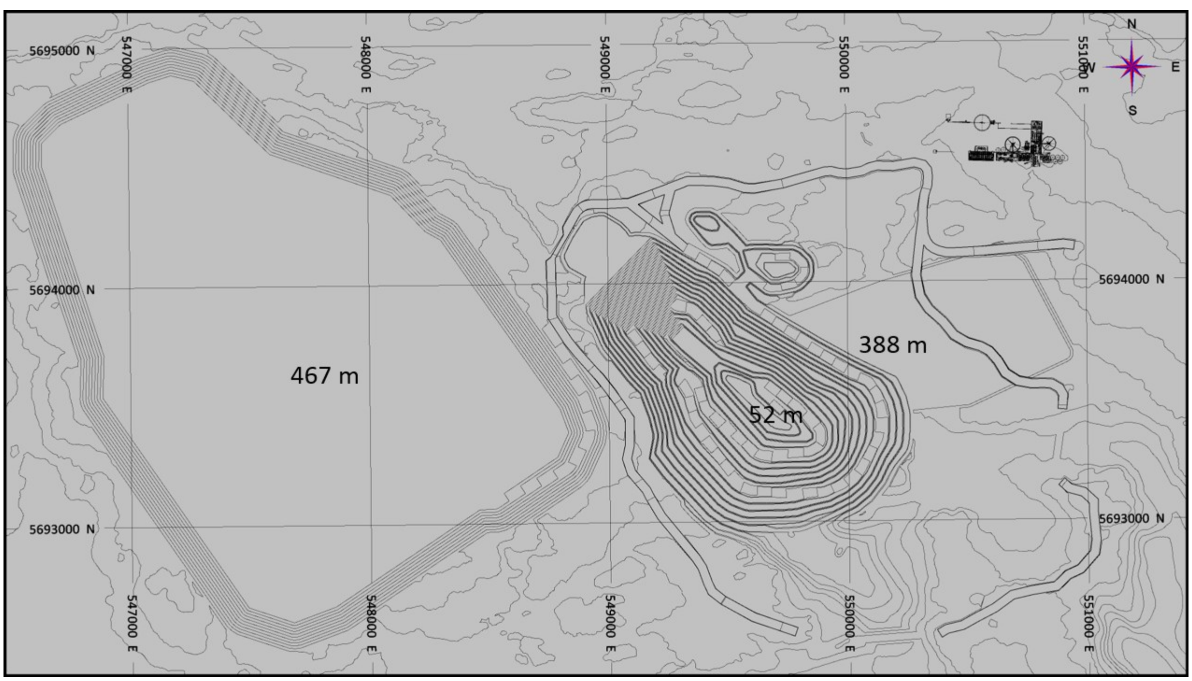
Source: AGP, 2021

Figure 16-28: End of Year 9



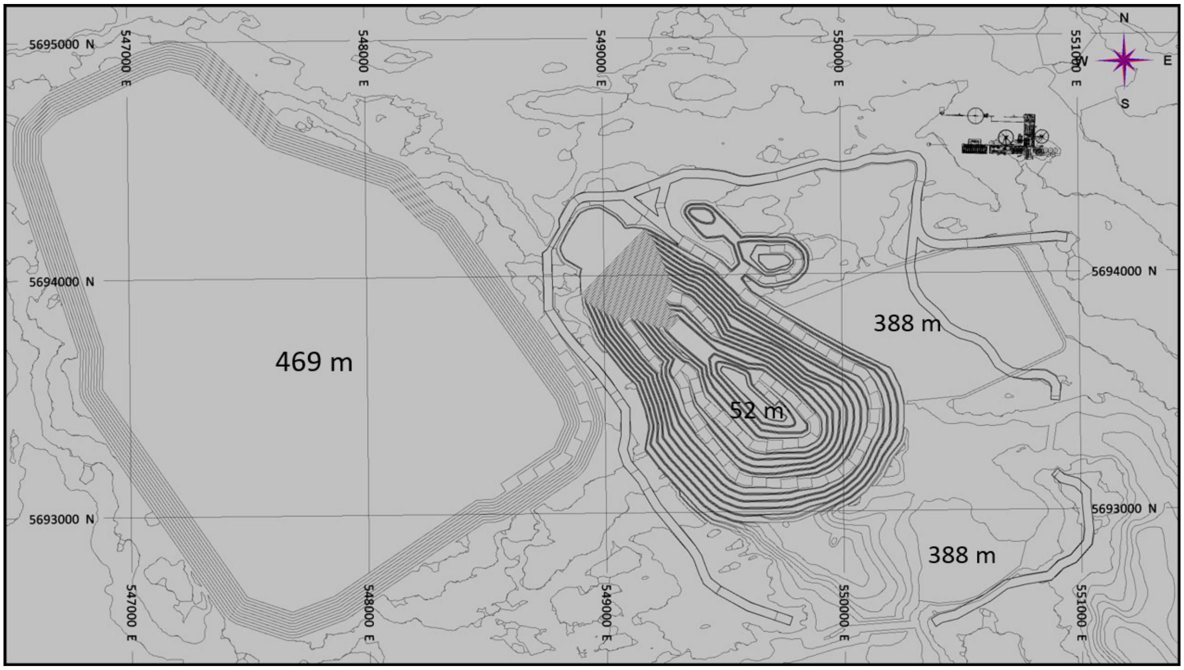
Source: AGP, 2021

Figure 16-29: End of Year 10 (End of Pit Mining)



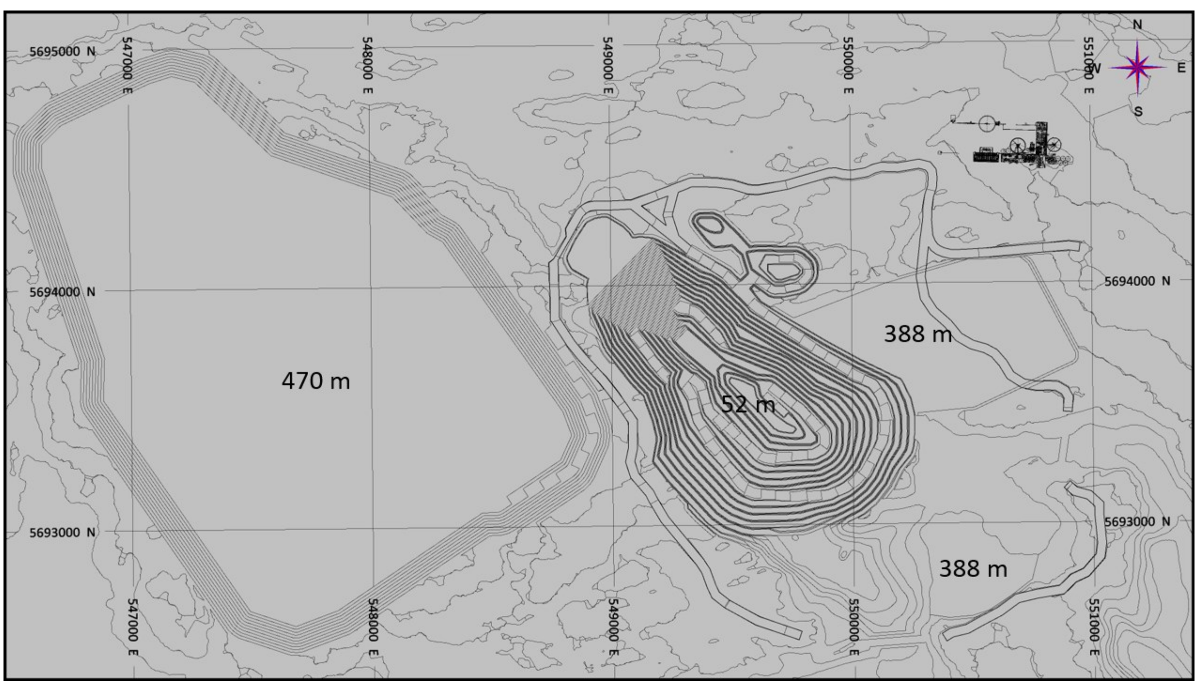
Source: AGP, 2021

Figure 16-30: End of Year 11



Source: AGP, 2021

Figure 16-31: End of Year 12 (End of Processing)



Source: AGP, 2021

16.11 Blasting and Explosives

Blast patterns vary for feed and waste to assist in mine productivity. Mill feed patterns will be 7.7 m x 6.7 m (spacing x burden) while waste patterns will be 7.7 m x 6 .7m (spacing x burden). Holes will be 12 m long plus an additional 1.3 m sub-drill for a total 13.3 m using a 251 mm bit.

Mill feed powder factors will be 0.30 kg/t, and the waste powder factor will be 0.30 kg/t. Only emulsion explosives will be used due to the expected wet conditions.

Additional drilling capability will be available with smaller 140 mm drills. They will supplement the drilling and provide the pre-shear drilling. When used in regular blast patterns the waste blast pattern will be 4.6 m x 4.0 m (spacing x burden) and for mill feed will be 4.8 m x 4.2 m (spacing x burden). The holes will be 12 m plus an additional 0.8 m sub-drill totaling 12.8 m. The powder factor for the smaller drill will be 0.27 kg/t for mill feed and 0.30 kg/t for waste.

Pre-shear holes will be 12 m deep, spaced 1.65 m apart and be separated from production blasts by 2 m. The powder charge will be 62 kg per hole in a decoupled manner.

16.12 Mining Equipment

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 251 mm bit. This provides the capability to drill 12 m bench heights in a single pass. The supplemental drill will also be a down the hole hammer drill with a 140 mm bit. This is capable of drilling 12 m benches but must add steel from its carousal.

The primary loading units will be 36 m^3 hydraulic shovels. Additional loading will be completed by 23 m^3 loaders. It is expected that one of the loaders will be at the primary crusher for the majority of its operating time. The haulage trucks will be conventional 240 t rigid body trucks.

A small fleet of equipment will be used for pre-construction activities including road building, WSF foundation preparation and cofferdam construction. This will mine in the proposed quarry area and the WSF footprint. The fleet will be comprised of 91 t rigid body trucks and 6 m^3 excavators. They will be initially matched with the smaller drills.

The support equipment fleet will be responsible for the usual road, pit, and dump maintenance requirements, but due to the climatic conditions expected, the support equipment 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.

16.13 Grade Control

Grade control will be completed with a separate fleet of reverse circulation (RC) drill rigs. They will drill the deposit off on a 10 m x 5 m pattern in areas of known mineralization, taking samples each metre. The holes will be inclined at 60°.

In areas of low-grade mineralization or waste the pattern spacing will be 20 m x 10 m with sampling over 12 m. These holes will be used to find undiscovered veinlets or pockets of mineralization.

These grade control holes serve to define the mill feed grade and mineralization contacts.

Samples collected will be sent to the assay laboratory and assayed for use in the short-range mining model.

Blasthole sampling will also be part of the grade control program initially to determine the best method for the planned Springpole operations.

17 RECOVERY METHODS

17.1 Summary

The process plant was designed using conventional processing unit operations. It will treat 30,000 tpd or 1,250 tph based on an availability of 8,059 hours per annum or 92%. The crusher plant section design is set at 75% availability and the gold room availability is set at 52 weeks per year including two operating days and one smelting day per week. The plant will operate with two shifts per day, 365 days per year, and will produce doré bars.

The recommended process flowsheet following grinding is flotation followed by cyanide leaching of both the concentrate and tailings streams. The 2019 updated PEA study (SRK, 2019) also recommended the use of flotation + concentrate/tailings leaching instead of the whole ore leach circuit option.

The plant feed will be hauled from the mine to a crushing facility that will include a gyratory crusher as the primary stage before being conveyed to the crushed ore stockpile. The crushed ore will be ground by a SAG mill, followed by a closed circuit of a ball mill with a hydrocyclone cluster. The hydrocyclone overflow with a P80 size of 100 mesh (150 µm) will flow to a three-stage flotation circuit including rougher flotation, rougher scavenger flotation, and cleaner flotation. Flotation tailings will report to the tailings leaching and CIP circuit. Flotation concentrate will report to a closed loop cyclone cluster and three regrind mills before reporting to the concentrate leach and CIP circuit.

Gold and silver leached in the CIP circuits will be recovered onto activated carbon and eluted in a pressurized Anglo American Research Laboratory (AARL) style elution circuit and then recovered by electrowinning in the gold room. The gold–silver precipitate will be dried in a mercury retort oven and then mixed with fluxes and smelted in a furnace to pour doré bars. Carbon will be re-activated in a carbon regeneration kiln before being returned to the CIP circuits. CIP tails will be treated for cyanide destruction prior to pumping to a final tailings thickener and pressure filter. Filter cakes will be hauled to the WSF for co-disposal with waste rock material.

The installed power for the process plant will be 59,482 kW and the power consumption is estimated to be 33.4 kWh/t processed. Raw water will be pumped from Birch Lake to a raw water storage tank. Potable water will be sourced from the raw water tank and treated in a potable water treatment plant. Gland water will be supplied from the raw water tank. Process water will primarily consist of reclaim water from the final tailings thickener and pressure filters. Reagents will include pebble lime, sodium cyanide, sodium hydroxide, copper sulphate pentahydrate, hydrochloric acid, sodium metabisulphite, activated carbon, flocculant, coagulant, collector (PAX), and frother (MIBC).

17.2 Overall Process Design

Gold occurs as fine grain and is associated with telluride minerals as well as silicates. The telluride-associated gold recovers well to flotation concentrate and further recovery improvement is seen when regrinding concentrate for improved liberation of fine-grained gold. Most of the remaining gold is associated with silicates and is recoverable by cyanidation.

The available testwork was thoroughly analyzed and several options of process routes were assessed in the initial stages of the Pre-Feasibility Study. Based on the analysis, a process route was chosen as the best suited for the testwork results and subsequent economic analysis for the material. The unit operations selected are typical for this industry.

The material is found to be highly variable, both in terms of ore hardness and metallurgical response. Variability within zones aligned with the planned mining sequence will be studied in greater detail ahead of the Feasibility Study phase, with consideration to be given to comminution properties, flotation, and leach response as well as filtration characteristics. The flowsheet selected has catered for a variable feed through selection of appropriate design criteria. The Project will utilize a capital cost-effective mill design, including a target grind (P_{80}) size of 150 µm, two stages of flotation, flotation concentrate regrind and leaching, flotation tails leaching, CIP, carbon elution and gold recovery. Leach-adsorption tailings will be treated for cyanide destruction, thickened, and filtered for co-disposal of filter cake with waste rock.

Key process design criteria include:

- throughput of 30.0 ktpd or 10.95 Mtpa
- crushing availability of 75%
- plant availability of 92% for grinding, flotation, leach, adsorption, desorption & cyanide reduction
- plant availability of 82.5% for tailings filtration
- plant availability of 91.7% for dry stacking

17.3 Mill Process Plant Description

The process design is comprised of the following circuits:

- primary crushing of run-of-mine (ROM) material
- grinding circuit comprising of a SAG mill followed by a ball mill with cyclone classification
- three stages of flotation: rougher flotation, rougher scavenger, and cleaner flotation
- flotation concentrate regrind and leaching
- flotation tailings leaching
- CIP adsorption from concentrate and tailings leaching stages
- acid washing of loaded carbon and AARL-type elution followed by electrowinning and smelting to produce doré
- carbon regeneration by rotary kiln
- cyanide destruction of tailings using O_2/SO_2 (INCO process)
- tailings thickening, filtration and waste storage facility
- effluent water treatment followed by a polishing pond before discharging to the environment

17.3.1 Plant Design Criteria

Key process design criteria for the plant are listed in Table 17.1.

Table 17-1: Key Plant Process Design Criteria

Design Parameter	Units	Value
Plant Throughput	tpd	30,000
Au Head Grade – Design	g/t Au	1.81
Silver Head Grade – Design	g/t Au	8.15
Crushing Plant Availability	%	75
Mill Availability	%	92
Filter Availability	%	82.5
Dry Stack Availability	%	91.7
Bond Crusher Work Index (CWi)	kWh/t	9.7
Bond Ball Mill Work Index (BWi)	kWh/t	13.5
Bond Abrasion Index (Ai)	g	0.11
JK Drop Weight Parameter Axb		72.2
Primary Crusher Type		gyratory
Material Specific Gravity	t/m³	2.68
SAG Mill Dimensions		32 ft dia. X 17 ft EGL
SAG Mill Installed Power	MW	9.5, with VSD
SAG Mill Discharge Density	% w/w	70
SAG Mill Ball Charge	% v/v	12
Ball Mill Dimensions (see Table 17-2 for revision)		24 ft dia. X 34.5 ft EGL
Ball Mill Installed Power (see Table 17-2 for revision)	MW	12.0
Ball Mill Ball Charge	% v/v	30
Circulating Load	%	350
Classification Cyclone Overflow Density	% w/w	35
Primary Grind size (P80)	µm	150
Rougher/Scavenger Flotation Cell Type		DFR
Rougher/Scavenger Flotation Residence Time	min	16
Cleaner Flotation Cell Type		DFR
Cleaner Flotation Residence Time	Min	20
Final Concentrate Mass Recovery	%	15
Concentrate Regrind Mill Type		IsaMill
Concentrate Regrind Mill Quantity		3
Concentrate Regrind Installed Power, each	MW	3
Concentrate Regrind Product Size (P80)	µm	17

Design Parameter	Units	Value
Flotation Concentrate Leach & CIP Tanks	#	3 + 7
Flotation Concentrate Leach + CIP Residence Time	h	18
Flotation Tails Leach & CIP Tanks	#	5 + 7
Flotation Tails Leach + CIP Residence Time	h	14
Elution Carbon Batch Size	t	13
Elution Strips per Week		26
Detox Residence Time	min	60
Detox WAD Cyanide Feed to Circuit	mg/L CN_{WAD}	250
Detox WAD Cyanide Discharge Target	mg/L CN_{WAD}	<5.0
Flotation Tails Thickener Underflow Density	% w/w	54
Final Tails Thickener Underflow Density	% w/w	55
Tailings Filter Type		Pressure
Tailings Filter Press Quantity		12
Filter Cake Moisture	% w/w	17

Testwork completed late in the PFS resulted in a change to mill design parameters. Going forward in the next phase of study, the design criteria will be updated to reflect this new information (Table 17-2). Contingency has been added to the PFS capital cost estimate to make provision for a larger ball mill, in alignment with the new hardness data. The operating cost estimate takes into account the larger ball mill motor.

Table 17-2: Revised Design Criteria for FS

Design Parameter	Units	Value
SPI	min	36
Bond Crusher Work Index (CWi)	kWh/t	9.9
Bond Ball Mill Work Index (BWi)	kWh/t	15.6
Bond Abrasion Index (Ai)	g	0.12
JK Drop Weight Parameter Axb		70.8
Crushed Ore Angle of Repose	degrees	37
Ball Mill Dimensions		25 ft dia. X 36 ft EGL
Ball Mill Installed Power	MW	13.5

Reagents in use in the process and the consumption rates are shown in Table 17-3.

Table 17-3: Reagent and Grinding Media Consumption

Reagent Description	Consumption g/t
Collector (PAX)	190
Frother (MIBC)	90
Quick lime	4261
NaCN	2364
Oxygen	1222
Activated carbon	46
NaOH	349
HCl	111
SMBS	1949
$CuSO_4$	110
SAG mill grinding media	150
Ball mill grinding media	260
Regrind mill media	52

17.3.2 Primary Crushing & Stockpiling

The crushing circuit is designed for an annual operating time of 6,570 h/or 75% availability.

Material will be hauled from the open pit to the primary crusher apron feeder that feeds the primary crusher at 1,667 t/h. The crushed material will be conveyed to a covered stockpile that will provide approximately 16 hours of live storage. The excess crushed material will allow routine crusher maintenance to be carried out without interrupting feed to the mill.

The mill feed stockpile will be equipped with apron feeders to regulate feed at 1,359 t/h into the SAG mill. Crushed material will be drawn from the stockpile by two apron feeders and will feed the mill circuit via the SAG mill feed conveyor.

The materials handling and crushing circuit will include the following key equipment:

- primary gyratory crusher
- mill feed apron feeders (equipped with VSDs)
- materials handling equipment

17.3.3 Grinding Circuit

The grinding circuit will consist of a SAG mill followed by a ball mill in closed circuit with hydrocyclones. The circuit is sized based on a SAG F_{80} of 150 mm and a ball mill product P_{80} of 150 µm. The SAG mill slurry will discharge through a trommel where the pebbles will be screened and recycled to the SAG mill via conveyor. Trommel undersize will discharge into the cyclone feed pump box.

The ball mill will be fed by the cyclone underflow and discharges through a trommel. The oversize will be screened out and discharged to a scats bunker. Trommel undersize will discharge into the cyclone feed pump box.

Water will be added to the cyclone feed pump box to obtain the appropriate density prior to pumping to the cyclones. Cyclone overflow will gravitate to the rougher flotation circuit via a trash screen.

The grinding circuit will include the following key equipment:

- 9.5 MW SAG mill (equipped with VSDs)
- 12 MW ball mill (to be updated to 13.5MW with recent testwork results)
- cyclone feed pump box
- classification cyclone cluster
- trash screen

17.3.4 Flotation Circuit

The flotation circuit will be comprised of three stages: rougher flotation, rougher scavenger flotation, and cleaner flotation. The equipment selected for all flotation stages are Woodgrove direct flotation reactors (DFRs).

Ore from the classification cyclones will feed four rougher DFR flotation cells. The rougher flotation circuit will require an addition of 110 g/t of PAX and 50 g/t of MIBC. Concentrate will be collected along the bank and will report to the cleaner circuit. Rougher tailings from the fourth cell report to the rougher scavenger circuit. The cleaner flotation circuit will include five DFR cells with reagent additions of 80 g/t PAX and 40 g/t MIBC. Concentrate from each of the cells will be collected and pumped to the cleaner flotation concentrate cyclone cluster and regrind mills before reporting to the flotation concentrate leach circuit.

The rougher scavenger circuit will include four DFR cells. Concentrate from each cell will be collected and sent to the cleaner flotation circuit. Tailings from the fourth cells in the bank will report to the flotation tails thickener. The flotation circuit will include the following key equipment:

- rougher DFR feed head tank
- rougher DFRs (4)
- rougher scavenger DFRs (4)
- rougher flotation concentrate pump box
- cleaner scavenger DFR head tank
- cleaner scavenger DFRs (5)
- cleaner flotation tails pump box
- cleaner flotation concentrate pump box
- flotation tails pump box

DFR flotation cells have been installed in similar processes with a proven track record. Pilot testing is planned ahead of the Feasibility Study phase with the objective of optimizing sizing of the cleaner circuit and downstream concentrate processing equipment.

17.3.5 Flotation Tails Thickener

Cleaner flotation and rougher scavenger flotation tailings will report to the flotation tails thickener. The 54 m diameter thickener will have a flocculant addition rate of 30 g/t and a coagulant addition rate of 30 g/t and target underflow density of 55% w/w before being sent to the tails leach tanks.

17.3.6 Cleaner Flotation Regrind Circuit

Cleaner flotation concentrate will be sent to the cyclone cluster. The cyclone cluster underflow will be split equally to three, 3MW regrind mills configured in open circuit. Regrind mill product at a P_{80} size of 15 µm will be combined with cyclone overflow and will report to the concentrate leach circuit.

In designing the installed mill power, a specific energy of 35.1 kWh/t was determined from testwork after the feed size was corrected to a P80 of 100 µm. The required power plus a 10% allowance for operational variability resulted in a total motor power of 8 MW. The three IsaMills will have a total installed power of 9 MW due to available equipment sizes (M10,000).

The cleaner regrind circuit will include the following key equipment:

- cyclone cluster
- M10,000 IsaMill regrind mills (3)
- mill feed and discharge pump boxes (6)

17.3.7 Flotation Concentrate Leach & Adsorption Circuit

The flotation concentrate leach-adsorption circuit will consist of one pre-aeration tank, three leach tanks and seven CIP tanks. The circuit will be fed by the flotation concentrate regrind circuit. The pre-aeration and leach tanks will be 3,300m^3 each and CIP tanks will be 1,000m^3 each, with a total circuit residence time of 30 hours at 30% w/w solids.

Oxygen will be sparged to each tank to maintain adequate dissolved oxygen levels for leaching at 20 ppm. Hydrated lime will be added to further refine the operating pH at a rate of 0.8 kg/t ROM feed. Cyanide solution will be added to the first and second leach tank. Fresh/regenerated carbon from the carbon regeneration circuit will be returned to the last tank of the CIP circuit and will be advanced counter-currently to the slurry flow by pumping slurry and carbon. Slurry from the last CIP tank will be pumped to the tails leach tanks.

The inter-tank screen in each CIP tank will retain the carbon whilst allowing the slurry to flow by gravity to the downstream tank. This counter-current process will be repeated until the loaded carbon reaches the first CIP tank. Recessed impeller pumps will be used to transfer slurry between the CIP tanks and from the lead tank to the loaded carbon screens that will be mounted above the acid wash columns in the elution circuit.

The flotation concentrate leach and carbon adsorption circuit will include the following key equipment:

- leach/CIP tanks and agitators
- loaded carbon screens
- inter-tank carbon screens
- carbon sizing screens

- carbon safety screens

17.3.8 Flotation Tailings Leach & Adsorption Circuit

The flotation tailings leach-adsorption circuit will consist of five leach tanks and seven CIP tanks. The circuit will be fed by flotation tailings thickener underflow together with barren solution from electrowinning cells. The leach tanks will be 5,470 m^3 and the CIP tanks will be 2,000 m^3 each, with a total circuit residence time of 21 hours at 48 % w/w solids.

Oxygen will be sparged to the first three tanks to maintain adequate dissolved oxygen levels for leaching at 20 ppm. Hydrated lime will be added to further refine the operating pH at a rate of 0.7 kg/t ROM feed. Cyanide solution will be added to the first and second leach tank. Fresh/regenerated carbon from the carbon regeneration circuit will be returned to the last tank of the CIP circuit and will be advanced counter-currently to the slurry flow by pumping slurry and carbon. Slurry from the last CIP tank will be pumped to the cyanide detoxification tanks.

The inter-tank screen in each CIP tank will retain the carbon whilst allowing the slurry to flow by gravity to the downstream tank. This counter-current process will be repeated until the loaded carbon reaches the first CIP tank. Recessed impeller pumps will be used to transfer slurry between the CIL tanks and from the lead tank to the loaded carbon screens that will be mounted above the acid wash columns in the elution circuit.

The flotation tails leach and carbon adsorption circuit will include the following key equipment:

- trash screens
- leach/CIP tanks and agitators
- loaded carbon screens
- inter-tank carbon screens
- carbon sizing screens
- carbon safety screens

17.3.9 Cyanide Detoxification

Leach-adsorption tails will feed the detoxification circuit at 47% w/w solids. The detoxification circuit will consist of two tanks configured in parallel, each with a residence time of approximately 60 min to reduce the CN_{WAD} concentration from 200 mg/L to less than 5.0 mg/L to comply with environmental requirements prior to deposition in the WSF.

Cyanide destruction will be undertaken using the SO_2/O_2 method. The reagents required are oxygen, lime, copper sulphate, and SMBS. The cyanide destruction tanks will be equipped with oxygen addition points and an agitator to ensure that the oxygen and reagents are thoroughly mixed with the tailings slurry. From the detoxification tanks, the tailings will report to the final tailings thickener.

The main equipment in this area will include:

- cyanide destruction tanks (2) and agitators (2)
- pump box

17.3.10 Tailings Dewatering

Detoxified tailings will be thickened and filtered before discharge to the WSF. The overflow of the 68 m diameter thickener will be reused as process water in the plant. Flocculant will be combined with the feed to the thickener to improve the settling rate of the material.

The underflow will be pumped into three filter feed tanks. Each tank will feed four pressure filters by dedicated feed pumps. The filtration plant will be designed for an annual operating time of 7,227 h/or 82.5% availability. A total of 10 filters will be installed and operating at any given time. The moisture content in the tailings will be reduced from 55% moisture content to 17% moisture content. Filtrate and filter wash water will be recycled within the process plant as process water.

Filter pressed material will then be loaded by a conveyor into a loadout bin and into haul trucks to be placed at the WSF. The method of tailings placement selected will be co-mingled tailings and waste rock placement.

The mining fleet delivering feed to the plant will be used to collect tailings for transport to the WSF. The plant layout was developed to take into account the flow of traffic required for feed delivery and tailings collection. Operational risks associated with this method of tailings placement in cold climates are discussed further in Section 18 of this Report.

The main equipment in this area will include:

- high-rate thickener
- overflow tank for process water storage
- underflow pumps
- filter feed tanks (3)
- filter feed pumps (10)
- pressure filters (10)
- belt feeders (10)
- loadout conveyor

17.3.11 Carbon Acid Wash, Elution, and Regeneration Circuit

Prior to the gold stripping stage, loaded carbon will be treated with a weak hydrochloric acid solution to remove calcium, magnesium, and other salt deposits that could render the elution less efficient or become baked on in subsequent steps and ultimately foul the carbon.

Loaded carbon from the loaded carbon recovery screen will flow by gravity to the acid wash column. Entrained water will be drained from the column and the column will be refilled from the bottom up with the hydrochloric acid solution. Once the column is filled with the acid, it will be left to soak, after which the spent acid will be rinsed from the carbon and discarded to the cyanide destruction tank.

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

The main equipment in this area will include:

- acid wash carbon columns (2)

The gold stripping (elution) circuit will use the AARL process. The elution sequence will commence with the injection of a set volume of water into the bottom of the elution column, along with the simultaneous injection of cyanide and sodium hydroxide solution to achieve a weak NaOH and weak NaCN solution. Once the prescribed volume has been added, the pre-soak period will commence. During the pre-soak, the caustic/cyanide solution will be circulated through the column and the elution heater until the working temperature is achieved.

Upon completion of the pre-soak period, additional water will be pumped through the recovery heat exchanger and elution heater, then through the elution column to the pregnant solution tank at a rate of 2.0 Bed Volumes (BV)/h. At this stage, the temperature of the strip solution passing through the column will be increased to 130°C and the gold will be stripped off the loaded carbon.

Strip solution will flow up and out of the top of the column, passing through the heat exchanger via the elution discharge strainers and to the pregnant solution tank.

Upon completion of the cool down sequence, the carbon will be hydraulically transferred to the carbon regeneration kiln feed hopper via a de-watering screen.

The stripping circuit will include the following key equipment:

- elution columns (2)
- strip solution heater (propane-fired) with heat exchanger (2)
- strip and pregnant solution tanks (6)

17.3.12 Carbon Reactivation

Carbon will be reactivated in two rotary kilns in parallel. Dewatered barren carbon from the stripping circuit will be held in a feed hopper. A screw feeder will metre the carbon into the reactivation kiln, where it will be heated to 750°C in an atmosphere of superheated steam to restore the activity of the carbon.

Carbon discharging from the kiln will be quenched in water and screened on a carbon sizing screen located on top of the CIP tanks to remove undersized carbon fragments. The undersize fine carbon will gravitate to the carbon safety screen, whilst carbon screen oversize will be directed to the CIP circuit.

As carbon is lost by attrition, new carbon will be added to the circuit using the carbon quench tank. The new carbon will then be transferred along with the regenerated carbon to feed the carbon sizing screen.

The carbon reactivation circuit will include the following key equipment:

- carbon dewatering screens (2)
- regeneration kilns (propane-fired) including feed hopper and screw feeder (2)
- carbon quench tanks (2)

17.3.13 Electrowinning and Smelting

Gold will be recovered from the pregnant solution by electrowinning and smelted to produce doré bars. The pregnant solution will pumped through eight electrowinning cells with stainless steel mesh cathodes. Gold will be deposited on the cathodes and the resulting barren solution will be pumped to the leach circuit.

The gold-rich sludge will be washed off the steel cathodes in the electrowinning cells using high-pressure spray water and gravitates to the sludge hopper. Sludge will be dewatered in a filter press and then transported manually using a tray to the mercury retort oven for mercury removal as well as simultaneous drying. Mercury collected will be sent off site for third-party processing.

Dried sludge will be removed from the oven the following day and combined with fluxes in a flux mixer before reporting to the smelt furnace. Once all the mixture has been added to the furnace and enough time has elapsed for the material to fully melt, the slag will be poured into a conical slag pot. The liquid metal will then be poured into molds on a mound tray. Cooled doré will then be cleaned, weighed, and stamped. The bars will be placed in a vault to await shipment to a refinery.

Dust collection will be provided in the gold room for smelting. Extraction fans are planned for the kiln, electrowinning cell, retort/drying oven, and smelting-furnace off gasses. All extraction fans will lead to a gas scrubbing system.

The electrowinning and smelting process will take place within a secure and supervised gold room.

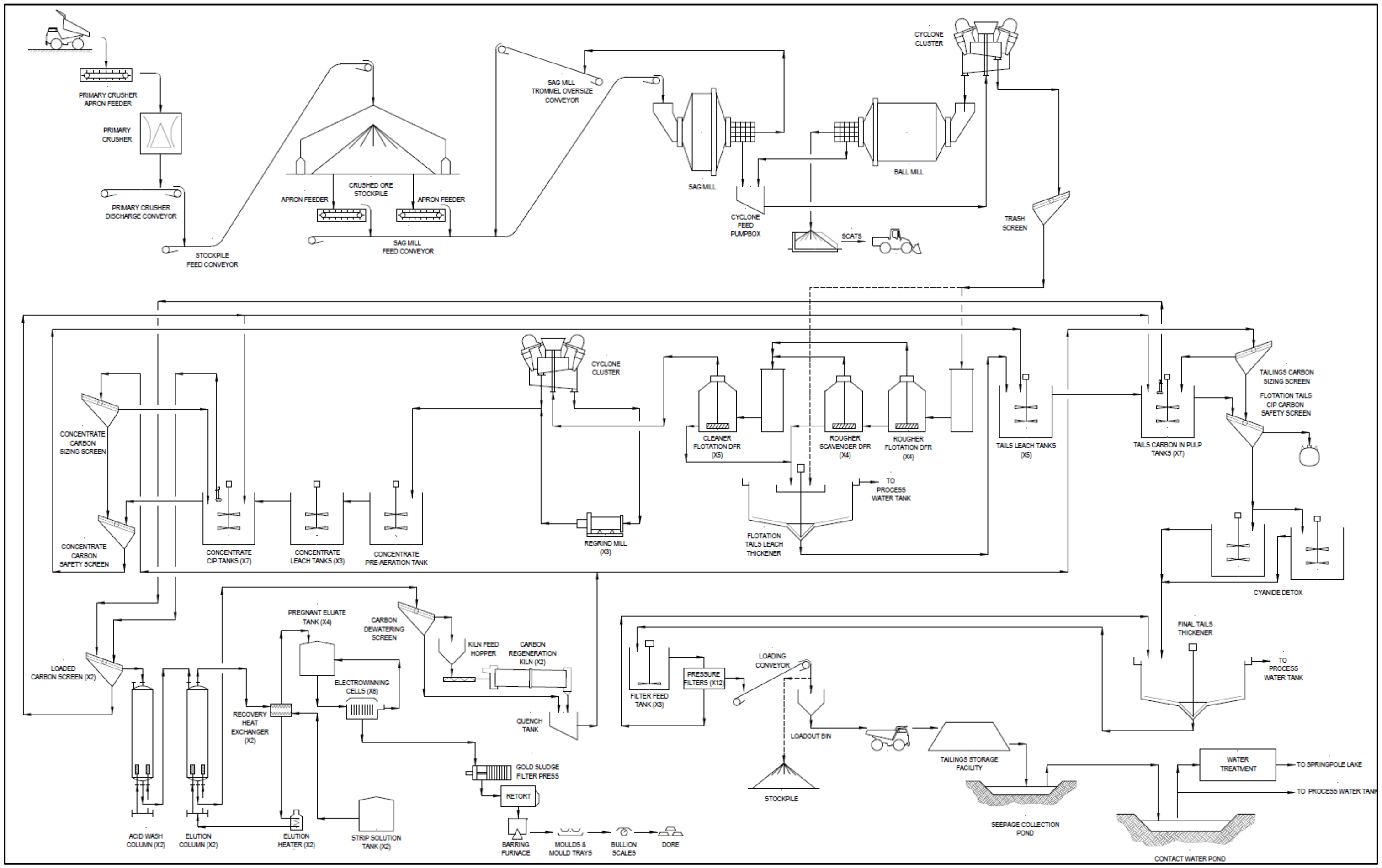
The electrowinning circuit and gold room will include the following key equipment:

- electrowinning cells with rectifiers (8)
- sludge pressure filter
- mercury retort
- flux mixer
- barring furnace with bullion moulds and slag handling system
- bullion vault and safe
- dust and fume collection and gas scrubbing system
- gold room security system

17.3.14 Flowsheet and Layout Drawings

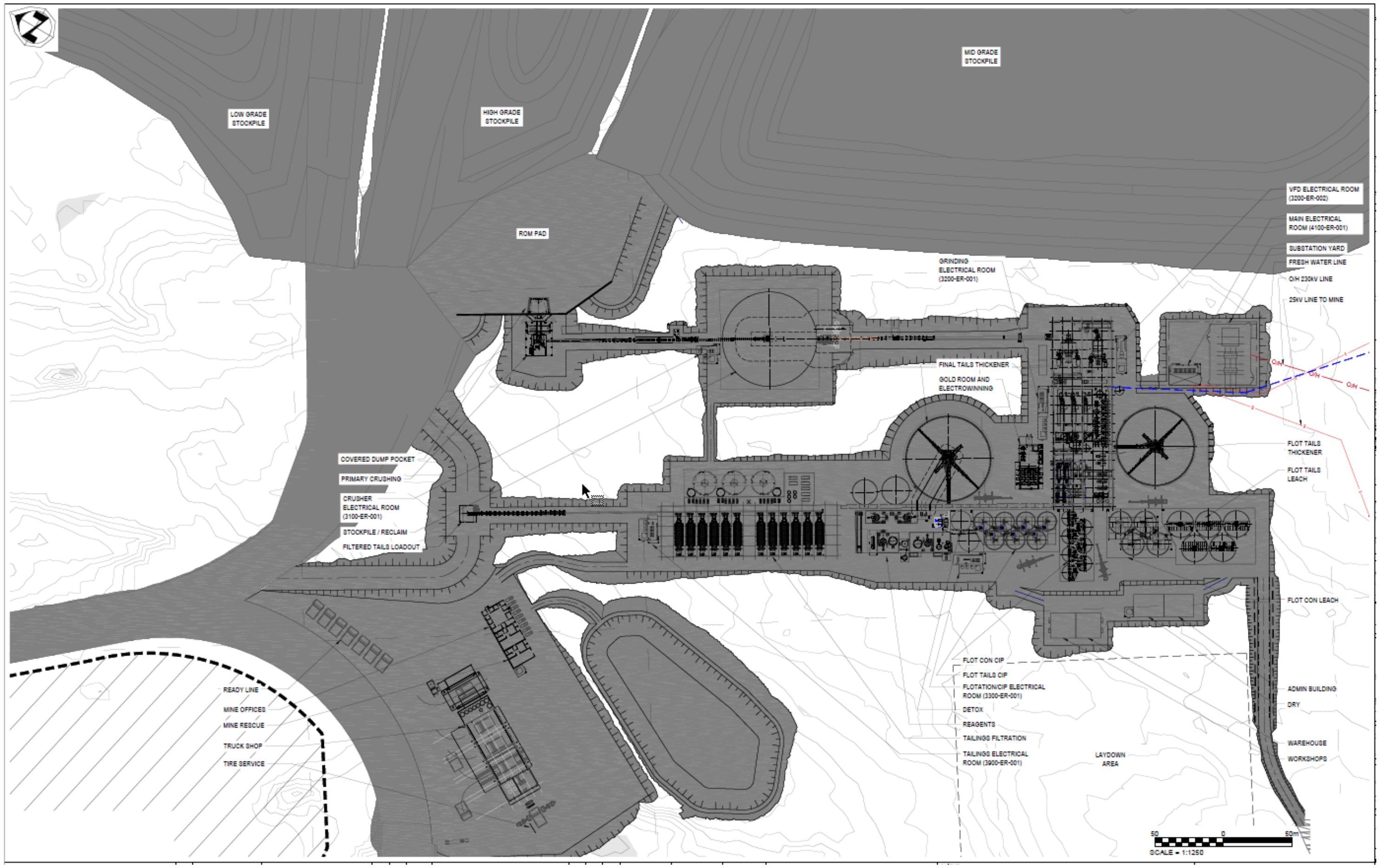
An overall process flow diagram showing the unit operations in the selected process flowsheet is presented in Figure 17-1. The proposed plant layout is shown in Figure 17-2.

Figure 17-1: Overall Process Flow Diagram



Source: SRK, 2021

Figure 17-2: Overall Plant Layout



Source: SRK, 2021

17.4 Reagent Handling and Storage

Each set of compatible reagent mixing and storage systems will be located within containment areas to prevent incompatible reagents from mixing. Storage tanks will be equipped with level indicators, instrumentation, and alarms to ensure spills do not occur during normal operation. Appropriate ventilation, fire and safety protection, eyewash stations, and safety data sheet (SDS) stations will be located throughout the facilities. Sumps and sump pumps will be provided for spillage control.

17.4.1 Pebble Lime

Pebble lime will be delivered in bulk and conveyed from the tanker to the pebble lime silo. Pebble lime will be extracted from the lime silo and fed into the lime slaking mill. A lime cyclone aids in maintaining a reagent stream density before it is sent to the lime mix storage tank. The lime distribution tanks will move lime solution from the lime mix storage tank to the flotation concentrate leaching circuit, cyanide detoxification, and flotation tailings leaching circuit.

17.4.2 Sodium Cyanide

Sodium cyanide will be delivered to site as briquettes in a bulk iso-tank, which will be directly connected to the sodium cyanide mixing tank.

After the mixing period is complete, cyanide solution will be transferred to the cyanide storage tank using a transfer pump. Sodium cyanide will be delivered to the flotation concentrate leach circuit, flotation tailings leach circuit, and elution circuit with dedicated dosing pumps.

17.4.3 Sodium Hydroxide

Sodium hydroxide (caustic soda) will be delivered in bulk as solution and offloaded from road tankers into the bulk storage tank. Dosing pumps will automatically deliver the reagent to the required locations—elution circuit and eluate tanks—to ensure the dosing requirements are met.

17.4.4 Hydrochloric Acid

Hydrochloric acid will be delivered in bulk as solution and offloaded from road tankers into the bulk storage tank. Hydrochloric acid will be delivered to the acid wash circuit using the hydrochloric acid dosing pump.

17.4.5 Copper Sulphate Pentahydrate

Copper sulphate pentahydrate will be delivered in solid crystal form in bulk bags. Process water will be added to the agitated copper sulphate mixing tank. Bulk bags will be lifted using a frame and hoist, and periodically a single bag will be placed on the copper sulphate bag breaker on top of the tank. The solid reagent will fall into the tank and will be dissolved in water to achieve the required dosing concentration.

Copper sulphate solution will be transferred by gravity to the copper sulphate storage tank, which will have a stacked arrangement with the mixing tank. Copper sulphate will be delivered to the cyanide detoxification circuit using the copper sulphate dosing pump. An extraction fan will be provided over the copper sulphate bag breaker/mixing tank to remove reagent dust that may be generated during reagent addition/mixing.

17.4.6 Sodium Metabisulphite (SMBS)

SMBS will be delivered in the form of solid flakes in bulk bags. Process water will be added to the agitated SMBS mixing tank. Bags will be lifted using a frame and hoist into the SMBS bag breaker on top of the tank. The solid reagent will fall into the tank and will be dissolved in water to achieve the required concentration. After the mixing period is complete, SMBS solution will be transferred to the SMBS storage tank using the SMBS transfer pump. SMBS will be delivered to the cyanide detoxification circuit using the SMBS dosing pump. An extraction fan will be provided over the SMBS mixing tank to remove SO_2 gas that may be generated during mixing.

17.4.7 Activated Carbon

Activated carbon will be delivered in solid granular form in bulk bags. When required, the fresh carbon will be introduced to the carbon quench tank, or directly to the final CIL tank.

17.4.8 Flocculant

Powdered flocculant will be delivered to site in bulk bags and stored in the warehouse. Two self-contained mixing and dosing systems will be installed, each including a flocculant storage hopper, flocculant screw feeder, flocculant preparation water heater, flocculant mixing tank, flocculant transfer pump, flocculant storage tank, and flocculant dosing pump. Powdered flocculant will be loaded into the flocculant storage hopper using the flocculant hoist. Dry flocculant will be pneumatically transferred into the wetting head, where it will be contacted with water.

Flocculant solution, at 0.50% w/v, will be agitated in the flocculant mixing tank for a pre-set period. After a pre-set time, the flocculant will be transferred to the flocculant storage tank using the flocculant transfer pumps. Flocculant will be further diluted just prior to the addition point. One system supplies to the flotation tails thickener and the other to the final tails thickener.

17.4.9 Coagulant

Coagulant will be delivered to site in totes and stored in the warehouse. Dosing pumps will deliver the reagent to the flotation tails thickener and final tails thickener as required.

17.4.10 Collector (PAX)

Potassium amyl xanthate (PAX) will be delivered in granular powder form in bulk bags. Raw water will be added to the agitated PAX mixing tank. Bags will be lifted using a frame and hoist into the PAX bag breaker on top of the tank. The solid reagent will fall into the tank and will be dissolved in water to achieve the required dosing concentration. PAX solution will be transferred by gravity to the PAX storage tank, which will have a stacked arrangement with the mixing tank.

The mixing tank will be ventilated using the PAX tank fan to remove any carbon disulphide gas. PAX will be delivered to the flotation circuit using the PAX dosing pump. Actuated control valves will provide the required PAX flow rates at several locations around the flotation circuit.

17.4.11 Frother (MIBC)

MIBC will be delivered as a liquid in IBC totes. It will be used as received and without dilution. Diaphragm-style dosing pumps will deliver the reagent to the required locations within the flotation circuit.

17.4.12 Gold Room Smelting Fluxes

Borax, silica sand, sodium nitrate, and sodium carbonate will be delivered as solid crystals/pellets in bags or plastic containers and stored in the warehouse until required.

17.5 Services & Utilities

17.5.1 Process/Instrument Air

High-pressure air at 750 kPa (gauge) will be produced by compressors to meet plant requirements. The high-pressure air supply will be dried and used to satisfy both plant air and instrument air demand. Dried air will be distributed via the air receivers located throughout the plant.

The tailings filtration area will have dedicated blowing and pressing compressors and air receivers.

17.5.2 Low Pressure Air

Low-pressure air for flotation will be supplied by air blowers to the rougher and cleaner flotation circuits.

17.6 Water Supply

17.6.1 Fresh Water Supply System

Fresh water will be supplied to a raw water storage tank. Raw water will be used for all purposes requiring clean water with low dissolved solids and low salt content, primarily as follows:

- gland water for pumps
- reagent make-up
- elution circuit make-up
- raw water is treated and stored in the potable water storage tank for use in safety showers and other similar applications
- fire water for use in the sprinkler and hydrant system
- cooling water for mill motors and mill lubrication systems (closed loop)
- fresh water is supplied from the fresh water source and as filtered contact water

17.6.2 Process Water Supply System

Overflow from the final tailings thickener and the flotation tailings thickener as well as filtrate and wash water from the tailings filters will meet the majority of the process water requirements. Raw water and contact water will provide any additional make-up water requirements.

Consideration should be given in future project phases to separate water circuits into cyanide and non-cyanide containing circuits to minimize the potential negative impact of cyanide recycle to flotation.

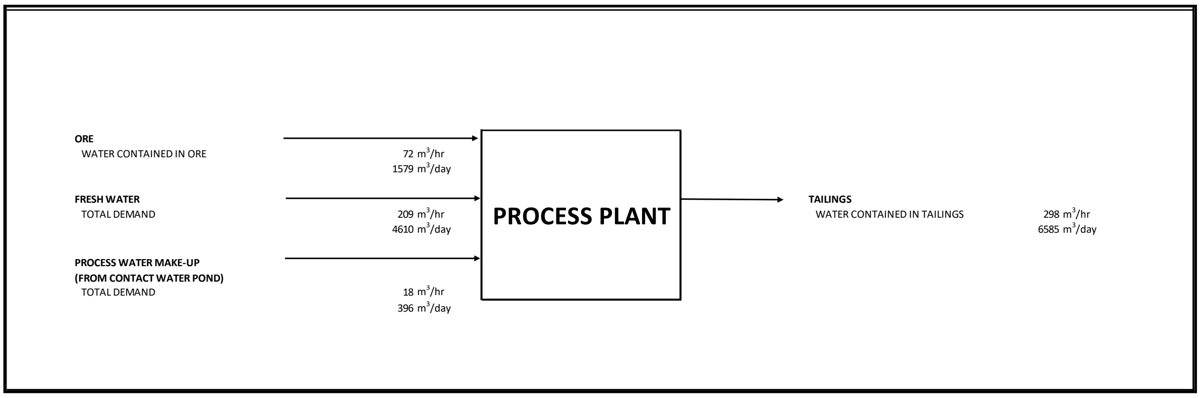
17.6.3 Gland Water

One dedicated gland water pump will be fed from the freshwater tank to supply gland water to all slurry pump applications in the plant.

17.6.4 Plant Water Demand

The overall projected plant water balance is shown in Figure 17-3.

Figure 17-3: Plant Water Balance



Source: AGP, 2021

17.6.5 Projected Energy Requirements

Electrical

The installed power for the process plant is estimated to be 58 MW, and power consumption is $2.76/t of material treated.

The installed power for the G&A areas will be 2.5 MW and power consumption will be $0.03/t treated.

Propane

Average propane demand is estimated at 33,000 L/d. The elution heaters and carbon regeneration kilns will account for approximately 65% of the propane demand. The balance is for the camp and process plant and infrastructure heating, ventilation, and air conditioning (HVAC) systems.

The cost of propane delivered to the Project location should be assessed against the cost of electrically powered equipment in future Project phases.

18 PROJECT INFRASTRUCTURE

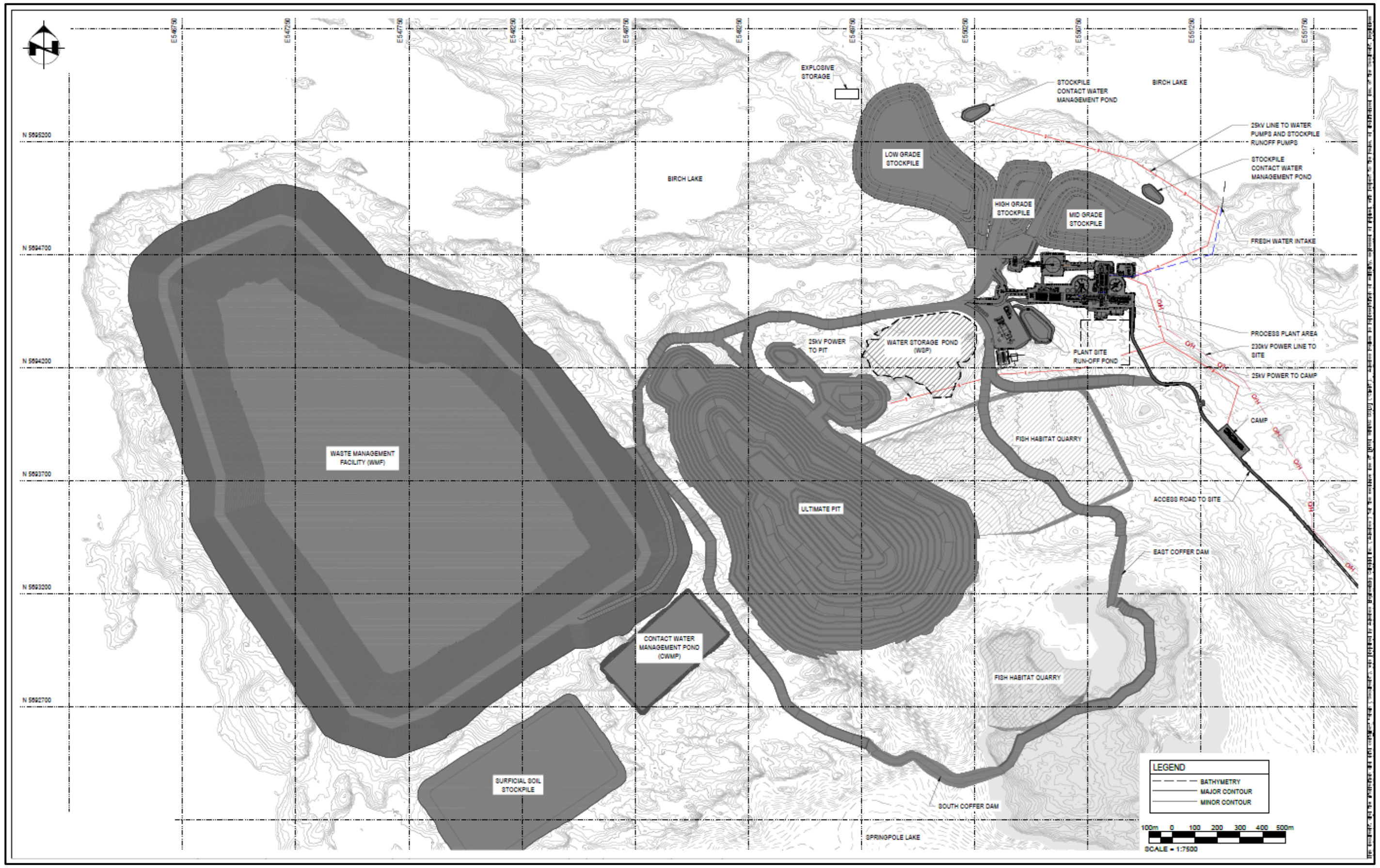
18.1 Introduction

Infrastructure contemplated in the 2020 PFS includes:

- Roads: main site access road, mine infrastructure access (MIA) road, administration in-plant access roads, cofferdam crest roads, waste rock and tailings dump haul road, explosives light vehicle access road, and ventilation raise and laydown light vehicle access road
- quarry
- open pit mining area
- WSF complete with co-located Contact Water Management Pond
- cofferdams A and B
- site main gate and guard house
- administration and dry building, training, first aid, change house and car park
- control room
- reagent storage building
- gold room
- assay laboratory and sample preparation area
- plant maintenance workshop and warehouse
- truck shop, parts and tool warehouse, truck wash building
- fuel facility, fuel storage and dispensing
- 230 kV overland, overhead transmission lines
- 230 kV tie-in to Provincial grid
- project site sub-station
- 25 kV power distribution
- freshwater intake, supply, and distribution
- contact water collection ponds
- contact water treatment plant
- raw water tank
- explosives magazine
- stockpile pads
- camp accommodations

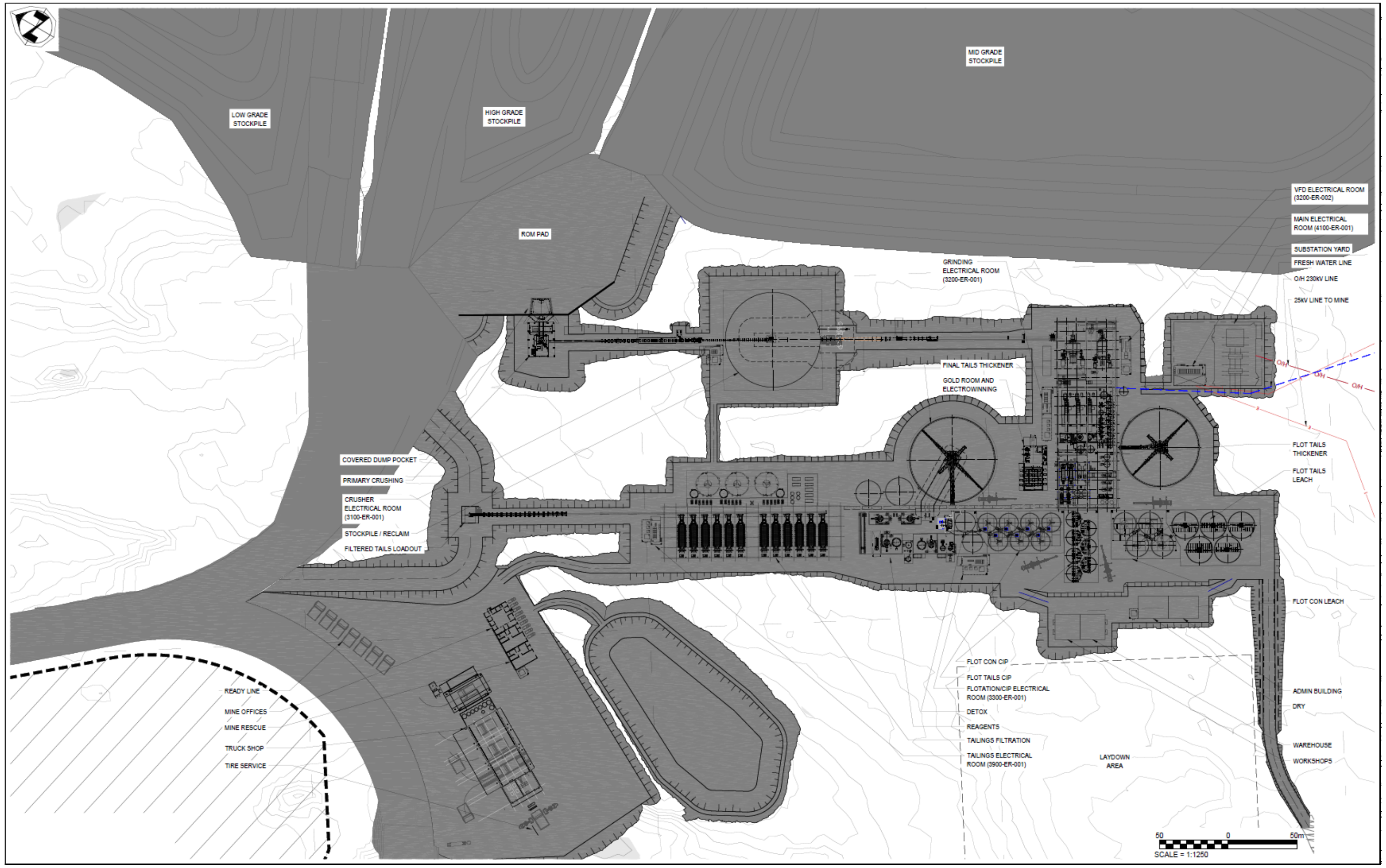
A layout of the proposed major infrastructure is included in Figure 18-1 and Figure 18-2.

Figure 18-1: Proposed Overall Site Layout Plan



Source:AGP, 2021

Figure 18-2: Proposed Infrastructure Layout Plan



Source: SRK, 2021

18.2 Access

18.2.1 Access to Site

Access to the Project area is described in detail in Section 5. This 2020 PFS envisages that the main access to the Springpole site will be the new Springpole Mine Road approaching from the southeast. This road is a final leg extension of the existing Wenesaga Road, a publicly-owned forestry road currently used for most of its length by EACOM timber for forestry activities. First Mining will be responsible for constructing and maintaining the extension from the termination of Wenesaga Road to Springpole site, which will be for First Mining's exclusive use. The main access road and location of the connection to the proposed road extension is shown above in Figure 18-1.

First Mining believes that the Project can play a meaningful role in encouraging the development of the road network in the area, with potential to connect the communities to the north with an all-season road that will provide access to the Municipality of Sioux Lookout, a major services hub for Northern Ontario. The PFS considers that the community access road may be completed prior to commencement of construction and is not part of the study.

18.2.2 On Site Roads

Within the Project area, the main access road will provide access to the process plant site, permanent camp, and will pass alongside the gatehouse. It will be consistent with a 40 km/hr design and posted speed and design criteria will be as follows: maximum longitudinal grade 8%, two lanes with a lane width of 4.0 m, road cross-slope of 3% and 1 vertical to 3 horizontal fill slope, 1 vertical to 2 horizontal back slope and 1 vertical to 3 horizontal side slopes.

Additionally, two in-plant and three maintenance roads will be constructed as follows:

- Mine Infrastructure In-Plant Access Road - a 100 m road running E-W connecting the Process Plant pad to the MIA
- Administration In-Plant Access Road - a short in-plant road that runs E-W connecting the Process Plant and Administration area
- Run-Off Pond Maintenance Road - a short N-S maintenance road connecting the MIA pad to the Run-Off Pond
- Stockpile Pad Maintenance Road - a N-S maintenance road, connecting the Stockpile Pad to the Process Plant area
- Substation Maintenance Road - a short E-W road, connecting the Process Plant to the Substation pad

In-plant and maintenance access roads will be consistent with a 30 km/hr design and posted speed limit and design criteria will be as follows: maximum longitudinal grade 12%, 3% road cross-slope and 1 vertical to 3 horizontal fill slope and 1 vertical to 2 horizontal back slope and 1 vertical to 3 horizontal side slope. In-plant roads will be dual lane 3.5 m wide each, and maintenance roads will be single lane, 5 m wide.

18.3 Waste Management

18.3.1 Design Basis

The primary design objectives for the WSF are secure confinement of the waste materials (both waste rock and tailings in a co-mingling manner) and protection of the regional groundwater and surface water during both mine operations and in the long-term (after closure). The design of the WSF and water management facilities has taken into account the following:

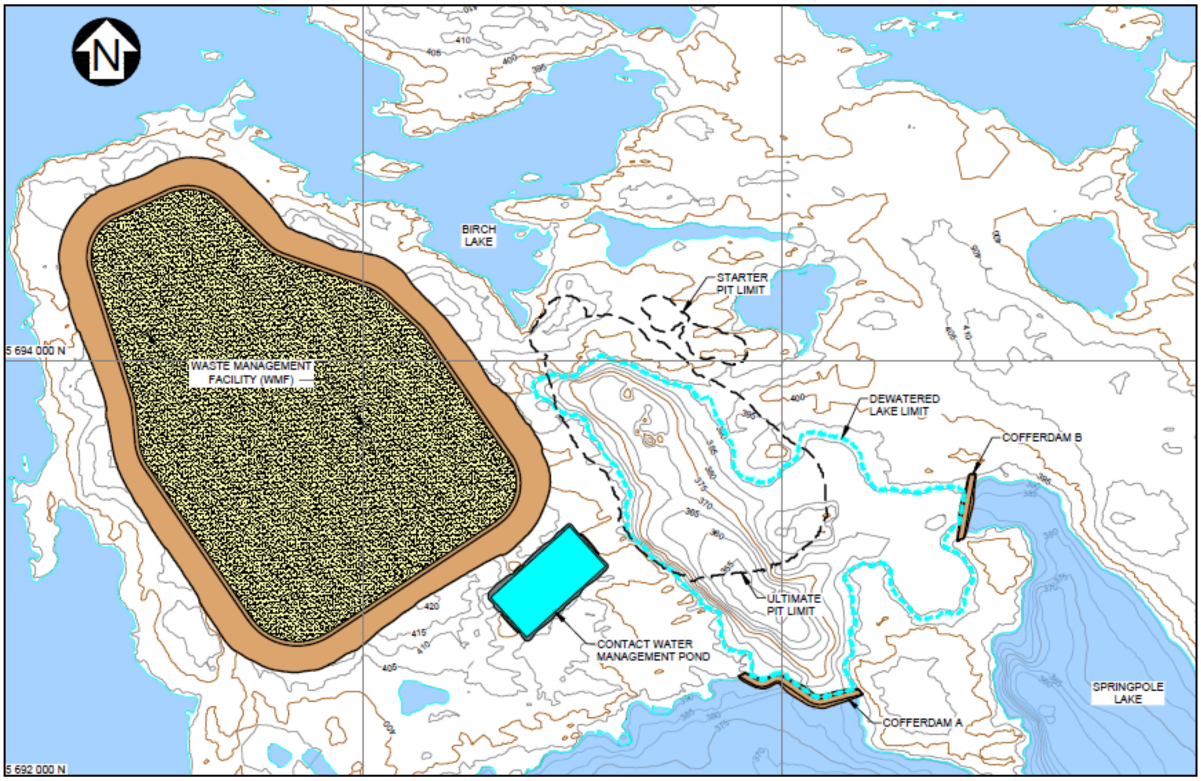
- optional geosynthetic lining of the facility to limit seepage (the requirement for a line will be reviewed during future design and environmental assessment phases)
- staged development of the facility over the life of the Project
- flexibility to accommodate operational variability in the filtered tailings (filter plant shutdowns and ore variability, along with placement during variable climate conditions)
- control, collection, and removal of water from the facility during operations for recycle as process water to the maximum practical extent

Approximately 117 Mm^3 of mine waste will be stored within the WSF, including 76 Mm^3 of tailings and 41 Mm^3 of PAG waste rock. The WSF is not expected to behave like a conventional mine waste facility (because of the large proportion of stored tailings) and will be constructed with perimeter embankments to provide stability for the overall structure. The construction of perimeter embankments will provide a number of benefits:

- Filtered tailings that do not meet moisture content or density targets will not have an impact on overall stability of the facility. The primary requirement for the filtered tailings will be the ability to transport the material to the facility and trafficability for subsequent placement.
- The perimeter embankment will provide freeboard for collection of annual run-off, along with temporary storage of design storm events.

The general arrangement of the WSF is shown on Figure 18-3.

Figure 18-3: Waste Storage Facility General Arrangement



Source: Knight Piésold, 2021

18.3.2 Hazard Classification

The design standards for the WSF are based on the relevant federal and provincial guidelines for construction of mining dams in Canada. The following regulations and guidelines were used to determine the dam hazard classification and suggested minimum target levels for some design criteria, such as the inflow design flood (IDF) and earthquake design ground motion (EDGM):

- Technical Bulletin – Application of Dam Safety Guidelines to Mining Dams (CDA, 2019)
- 1990 Ontario Lakes and Rivers Improvement Act (LRIA, 2011)

The WSF has been classified as Very High under both CDA guidelines and the LRIA. The recommended IDF during operations is defined as 2/3 between the 1/1000-year return period flood and the probable maximum flood (PMF) for a very high dam classification. A 72-hour event will be considered, given the WSF will not be constructed with a spillway and will need to temporarily 'store' a flood event while dewatering and treatment (if required) of collected run-off is completed. EDGM parameters have been determined for the WSF using estimates from the Natural Resources Canada (NRCan) seismic hazard calculator. The design earthquake is characterized as halfway between the 2,475-year and the 10,000-year return period seismic events for a Very High dam classification. The subsequent peak ground acceleration (PGA) is 0.056 g.

18.3.3 Tailings Characteristics

As part of the PFS, preliminary geotechnical testing was completed on a sample of tailings to determine its physical and mechanical properties. The following tests were completed; hydrometer, Atterberg limits, standard Proctor, hydraulic conductivity, and triaxial tests. The results are shown in Table 18-1.

Table 18-1: Physical and Mechanical Tailings Properties

Description	Value
Specific Gravity	2.81
Sand Content (> 0.06 mm; %)	32
Fines Content (< 0.074mm; %)	68
Clay Content (< 0.002mm; %)	10
Liquid Limit (%)	26
Plastic Index (%)	Non-Plastic
USCS classification	ML (Inorganic Silt)
Hydraulic Conductivity (m/s)	1.57E-07
Standard Proctor – Max Dry Density (g/cm^3)	1.895
Standard Proctor – Optimum Moisture Content (%)	13.5
Consolidated Undrained Triaxial – Friction Angle (o)	31

The tailings are classified as a non-plastic inorganic silt with a low permeability when compacted at the proposed filtered moisture content. The consolidated undrained shear strength is typical for an inorganic silt.

18.3.4 Facility Design

The WSF footprint will be logged and cleared for foundation preparation and embankment construction. Basin preparation will include removal of overburden material from low points within the topography and placement over any rock outcrops, prior to placement of a geosynthetic liner (the requirement for a liner will be reviewed during future design and environmental assessment phases). Overburden materials will be removed beneath the embankment foundations prior to fill placement. The focus of material removal is expected to be within low points around the perimeter of the WSF, where small waterbodies currently exist. It is assumed that an average 5 m of overburden removal will be required over the footprint of the embankment.

The WSF will initially be constructed as two cells, the central cell, and the north cell. The construction of two cells will minimize disturbance during the pre-production period, as well as maximize the use of NAG waste rock generated from open pit mining for construction of the perimeter embankments. A foundation drainage network will be developed within the base of the facility using selective placement of waste rock. See Section 18.5 for details on water management.

The WSF central cell construction will begin in Year -2 and will require approximately 4 Mm^3 of fill material for construction to El. 425 m. The embankment will be constructed with 3H:1V upstream slopes and 2H:1V downstream slopes. It is expected that construction of the central cell will be completed primarily with fill material sourced from a local borrow (developed within the basin of the north cell), while NAG waste rock produced from initial pit stripping will be prioritized for construction of site access roads, low grade-ore stockpile pads, cofferdams, etc. during Year -2. The remainder of the central cell embankment will be completed in Year -1, primarily with NAG rockfill from open pit stripping delivered by the mine haul truck fleet.

The WSF north cell construction will begin in Year 1 and will require approximately 3.5 Mm^3 of fill material for construction to El. 425 m. The embankment will be constructed with 3H:1V upstream slopes and 2H:1V downstream slopes, similar to the central cell. The north cell embankment will be constructed with NAG rockfill sourced from open pit stripping.

The WSF will be constructed as a single (combined) cell from Year 2 onwards. NAG rock generated from open pit mining will be used to construct downstream raises of the embankment through Year 6 to El. 470 m (maximum dam height of approximately 70 m). The embankment will continue to be constructed with 3H:1V upstream slopes and 2H:1V downstream slopes. A total of approximately 61 Mm^3 of fill material from mining operations is required for construction of the embankment (in addition to that used for construction of the central and north cells).

18.3.5 Waste Placement

Material will be hauled from the mine and the plant for placement in the WSF. Waste from the pit will be determined by assay methods if it is NAG or PAG. NAG material will be used to develop the outside wall or buttress of the facility. Normal dozer support will be used in the construction of the berm with occasional compactor assistance.

PAG material will be placed directly in the interior of the WSF as per a normal waste dump. Tailings, brought from the plant on a backhaul, will be placed together with the PAG material. The compactor will be used as required to ensure the material is compacted.

18.3.6 Monitoring

Instrumentation and monitoring will be required to assess embankment performance. Vibrating wire piezometers will be installed to monitor pore pressure within the embankment fill materials and slope inclinometers and survey monuments will be installed to monitor slope movement and deformation.

18.3.7 Water Management

The surface of the filtered tailings and PAG waste rock within the WSF will be graded to encourage flow to defined sumps/pumping points. The collected surface water will be directed to a contact water management pond (CWMP) that will be located to the southeast of the WSF. The water stored in the CWMP will be used as the primary source of make-up water for the process plant. Any water in excess to the process plant requirement (if any) will be directed to the WTP and released to Springpole Lake in accordance with regulatory requirements.

Water that infiltrates the surface will fill any void space remaining within the waste materials and may end up contributing to elevation of the phreatic surface at the base of the facility (the phreatic surface may also be elevated because of loading from subsequent waste placement if the material is placed at or near saturation). A drainage system will include pumping stations (drainage ponds) with submersible pumps installed within screened high-density polyethylene (HDPE) pipe to facilitate collection of water and transfer to the CWMP. The HDPE pipe will be extended as the waste surface rises. The primary drainage ponds and pumping systems will be installed at topographical low points within the WSF footprint.

18.4 Cofferdams

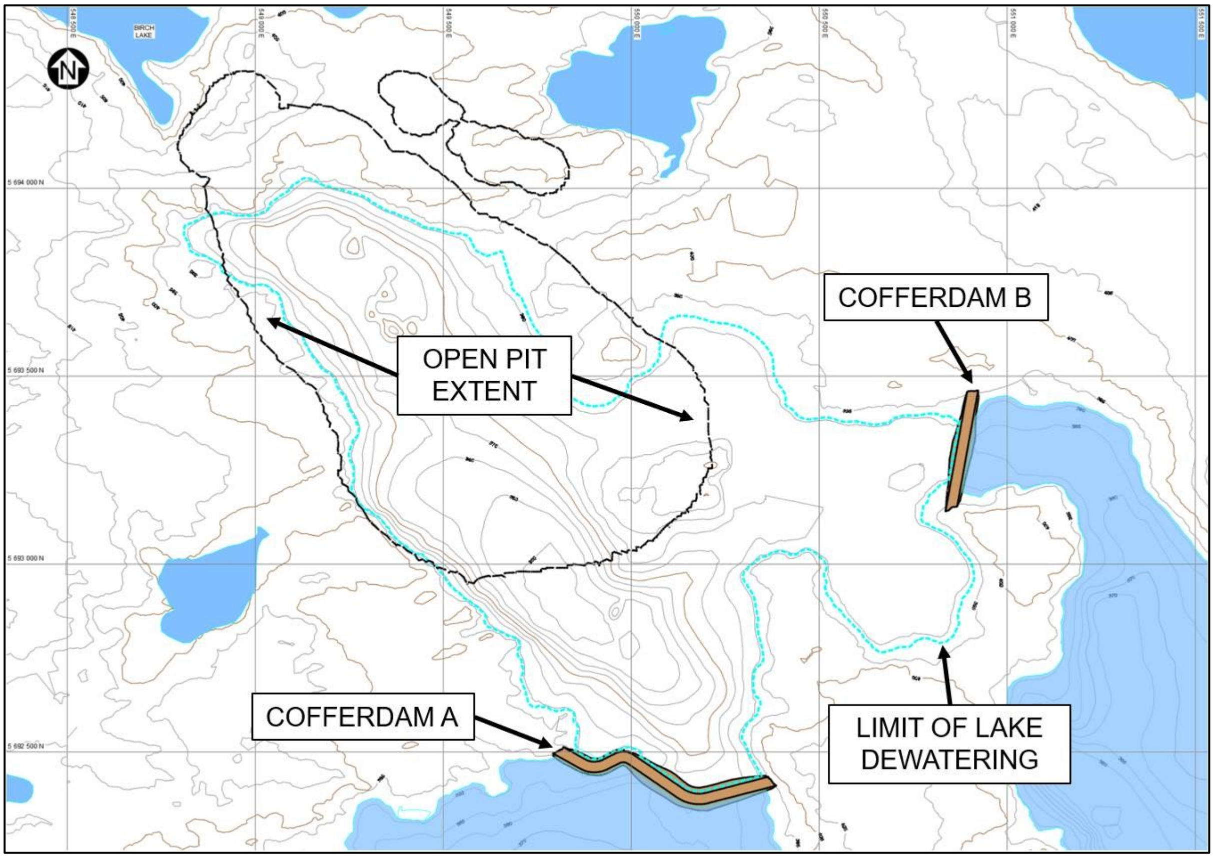
18.4.1 Design Basis

Two cofferdams (Cofferdam A and Cofferdam B) will be constructed across Springpole Lake, as shown in Figure 18-4, to isolate the open pit from Springpole Lake during operations (dewatering and mining). The design of the cofferdams will address the following requirements:

- achieve freeboard above the lake level that provides protection during storm events, along with protection from ice build-up that may occur in the winter
- maintain cofferdam stability during and following bay drawdown (meeting or exceeding regulatory factor of safety requirements)
- establish a hydraulic barrier following bay drawdown

The closure and reclamation objectives for the Project will include returning the dewatered region to Springpole Lake through construction of a spillway within the cofferdams to allow natural refilling of the basin.

Figure 18-4: Location of Cofferdams A and B, and Limits of Bay Dewatering



Source: Knight Piésold, 2021

18.4.2 Hazard Classification

The design standards for the cofferdams are based on the relevant federal and provincial guidelines for construction of mining dams in Canada. The following regulations and guidelines were used to determine the dam hazard classification and suggested minimum target levels for some design criteria, such as the IDF and EDGM:

- Technical Bulletin – Application of Dam Safety Guidelines to Mining Dams (CDA, 2019)
- 1990 Ontario Lakes and Rivers Improvement Act (LRIA, 2011)

The cofferdams have been classified as Very High under both CDA guidelines and the LRIA . The recommended IDF during operations is defined as 2/3 between the 1/1000-year return period flood and the PMF for a Very High dam classification. A 72-hour event will be considered, given the cofferdams will not be constructed with spillways (during operations) and will need to provide temporary 'storage' as Springpole Lake responds to the flood event. EDGM parameters have been determined for the cofferdams using estimates from the Natural Resources Canada (NRCan) seismic hazard calculator. The design earthquake is characterized as halfway between the 2,475-year and

10,000-year return period seismic events for a Very High dam classification. The subsequent PGA is 0.056 g.

18.4.3 Embankment Design

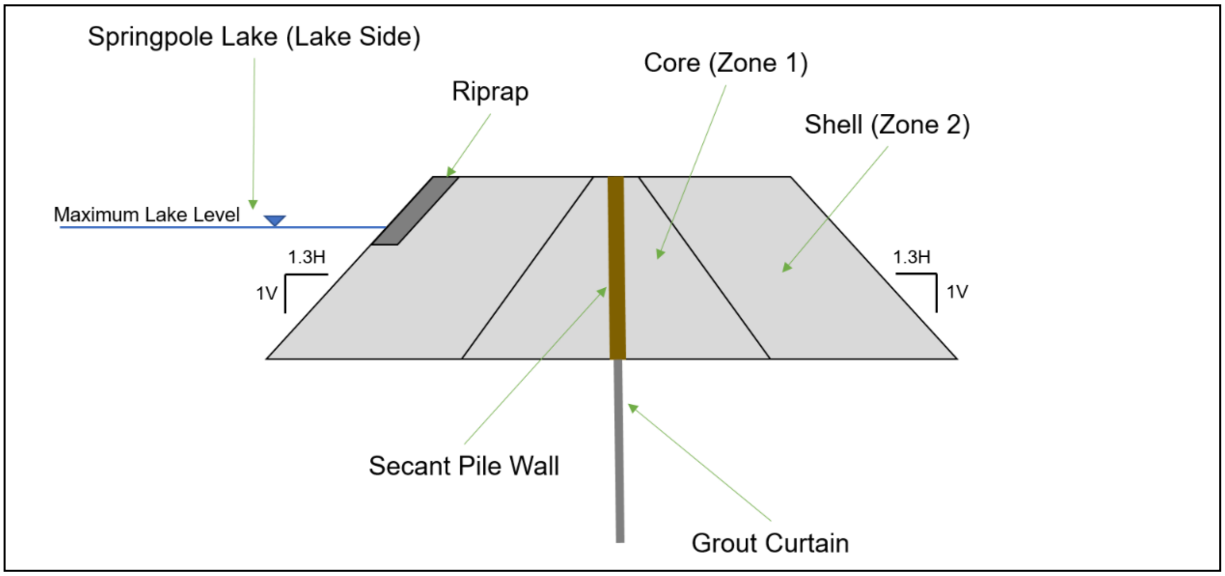
The cofferdams will initially be constructed as homogeneous NAG rockfill embankments using material sourced from initial stripping of the open pit. The embankment will be advanced from the shoreline across Springpole Lake with a crest width of 28 m to allow mine fleet haul trucks sufficient room to turn (3-point) and dump. The upstream and downstream slopes will not be controlled during placement of embankment fill and are assumed to settle at an approximate 1.3H:1V angle of repose. The cofferdams will have a freeboard of 5 m, as measured from the dam crest (El. 396 m) to the approximate elevation of Springpole Lake (El. 391m). The allowance of 5 m is based primarily on protection for ice heaving at the upstream dam crest in addition to wave runup. The cofferdams will not be constructed with spillways and therefore the freeboard allowance will also need to provide flood protection during the IDF. The embankment will be constructed of three primary material zones, as shown on Figure 18-5:

- Zone 1 is a 2-inch minus rock fill material used for construction of the central core zone of the embankment. Zone 1 material is required though the core of the cofferdam to facilitate drilling and reduce cement bentonite slurry loss during the secant pile wall installation.
- Zone 2 material is a general rock fill material used for construction of the embankment shell.
- Riprap will be placed in a 2 m thick layer on the upstream faces of the cofferdams to provide erosion protection.

A grout curtain will be developed within the foundation following placement of embankment rockfill. The grout curtain will reduce seepage through the top of the bedrock underlying the cofferdams and reduce pore pressure at the downstream embankment toe following dewatering of the open pit side of Springpole Lake. The grout injection depth into the bedrock is assumed to be 5 m for the current study. The grout curtain will be drilled and grouted prior to construction of the secant pile wall (SPW).

A SPW will be installed within the embankment rockfill to limit seepage and isolate the area of the proposed open pit from Springpole Lake. The SPW drillholes will be backfilled with a cement bentonite mixture in an alternating pattern, with overlap between adjacent piles (i.e., drilling of every third pile, then returning to establish connection when the perimeter piles have set). The SPW will be installed in Zone 1 material to minimize slurry loss. The SPW will extend into competent bedrock where it will overlap with the grout curtain by a minimum 0.2 m. Concrete cut-off walls will be constructed at the abutments of the cofferdams (tie-in points to land) where the depth of cut-off is less than 1.5 m.

Figure 18-5: Typical Section – Cofferdam Construction



Source: Knight Piésold, 2021

18.4.4 Monitoring

Instrumentation and monitoring will be required to assess embankment performance. Vibrating wire piezometers will be installed to monitor pore pressure within the embankment fill and foundation materials and slope inclinometers and survey monuments will be installed to monitor slope movement and deformation.

18.4.5 Environmental

A turbidity barrier will be installed on both the upstream and downstream side of the cofferdam alignment prior to placement of embankment fill material. It is expected that the upstream turbidity barrier will remain in place following construction, isolating Springpole Lake from any sediment laden run-off from the embankments.

18.5 Water Management

18.5.1 Surface Water Management

Several water control structures were designed for surface water management by adopting the Best Management Practices for the Project. All structures are to be regularly inspected to ensure continued functionality and performance. Key water collection infrastructure will include:

- CWMP
- WSP
- WSF
- drainage diversion channels and ditches

The stormwater management systems design is intended to divert and intercept the surface run-offs outside of the Project's main functional areas (open pit, process plant area, WSF, and low-grade ore stockpile) and prevent surface water from entering the mine site, thereby decreasing the quantity of contact water. These surface run-offs will be considered non-contact water. The diversion works for managing the non-contact water will include perimeter and diversion channels, water capture, crossing and water release structure, and fresh-water ponds, to be sized for a nominally 1-in-10 year rainfall.

Streams where qualities may potentially be affected by the mine components or project activities will be treated as contact water. The following control techniques have been envisaged to manage the mine contact water (store, treat, and release to the environment), prevent stormwater damage to the mine facilities and infrastructure, and provide make-up water for the process plant:

- CWMP that will be located southeast of the WSF will collect and provide recycled water for ore processing to reduce the freshwater demand in the process plant
- collecting contact water from the WSF, open pit, low grade ore stockpile, and plant site in lined collection/storage ponds (more details on the main collection ponds are provided below)
- treating the contact water stored in the WSP and discharging it to Springpole Lake

18.5.2 Waste Storage Facility (WSF)

The surface of the filtered tailings and PAG waste rock within the WSF will be graded to encourage surface flow to defined sumps/pumping points. The collected surface water will be directed to the CWMP. The water stored in the CWMP will be used as the primary source of make-up water for the process plant. Any excess water (if any) will be directed to the WTP and released to Springpole Lake.

18.5.3 Plant Site

Stormwater run-off that does not come in contact with the plant site's pads, is considered clean, and will be directed away from the plant site through diversion ditches.

A plant site run-off pond will be located in a low-lying area south of the plant site adjacent to the MIA. This pond will be single lined with a HDPE liner and will collect the run-off from the plant site. The pond is sized to contain the 100-year, 24-hour storm run-off event from the Process Plant, ROM pad, Primary Crusher and MIA pads.

The contact water will be collected and conveyed by gravity through a series of ditches and culverts to the plant site run-off pond for temporary storage. The water collected in this pond will be pumped directly to the WTP, where it will be treated prior to being released to Springpole Lake in accordance with regulatory requirements.

18.5.4 Mine Pit Dewatering

Pit dewatering will deal with the surficial run-off water due to rainfall and snowmelt events as well as expected seepage in the pit walls as the mine progresses deeper.

Pumping will be accomplished with staged electric pumps in the pit to a surface transfer pond. Two pumps staged will pump from the pit. From the surface pond a single pump will pump water horizontally to the water storage pond. Some of this water will be used for plant operations with the excess treated (if required) and discharged in accordance with regulatory requirements.

Pumping rates are initially estimated at 1.8 Mm^3 per year but rise to 6.2 Mm^3 per year in Year 9. Pumping will be a 24 hour, seven days per week operation for the life of the mine.

18.5.5 Water Storage Pond (WSP)

The WSP with a 350,000 m^3 capacity will be designed to contain and temporarily store the surge volumes from the pit dewatering flows and low-grade ore stockpile run-offs (collected in a few small ponds around the low-grade ore stockpile). A small portion of the water stored in this pond will be used to supplement the process plant make-up water. The remaining water in the pond will be treated by the WTP, and then released to Springpole Lake.

The WSP will be constructed in the dewatered lake adjacent to the Starter pit, located to the south-west of the process plant. Two berms for containment of the WSP will be built from excess waste rock during the pre-stripping phase of the Project.

The WSP will not be lined, however, the lake sediment at the base will remain intact to act as a low-permeability liner. It is assumed that any seepage from the WSP will drain into the pit and be recirculated to the pond through the pit dewatering efforts.

18.6 Site-wide Water Balance

A site-wide water balance was completed to estimate the quantity of mine site contact water expected to be managed during the operational phase of the Project to support the Pre-Feasibility Study. The site-wide water balance used a model developed in AuSim on a monthly timestep basis for the operation phase of the mine (life of mine excluding the pre-production phase, i.e. years -2 and -1). For stochastic analysis, AuSim uses the Monte Carlo method to generate monthly precipitation and evaporation lake time series for the water balance model.

The water balance model considered the precipitation and groundwater gains, and evaporation and infiltration losses (where applicable) of the following mine components: Open Pit, WSF, Process Plant and associated Infrastructure, Camp Site and Sewage Treatment Facility, Low Grade Ore Stockpile, and WSP. The site-wide water balance study provides a water management strategy focused on the following main objectives:

- estimating the effluent flows of the main mine components
- estimating the capacity of the WTP
- estimating the volume of the water storage pond
- estimating the discharge water quantity to the environment (Springpole Lake)

The model's main inputs were obtained from current meteorology reports and other available information regarding hydrogeology, operation strategy, open pit water management, and WSF water management (PFS level).

The model considered over 100 simulations to obtain acceptable probabilistic estimates of the effluent flows for the mean (monthly average) and 95% non-exceedance probability (wet condition) scenarios described below:

- mean (monthly average): the average monthly flow of all simulations

- 95% non-exceedance probability (p95%): maximum monthly flow with 5% of risk considering all simulations. This means that there is a 5% probability that the reported value will be exceeded in one month of the simulation period. It is important to note that the p95% scenario may occur for each mine component independently. For instance, it is possible that while the WSF experiences a wet condition (p95%), another component, e.g. the low-grade ore stockpile experiences the Mean scenario.

It should be noted that year 1 to year 10 of the mine operations were modelled separately from year 11 to year 12 because the pit dewatering, which results in the largest flows, occurs only from year 1 to year 10. Therefore, the last two years of the mine operations (year 11 and year 12) were modelled independently.

The following are the main conclusions derived from the site-wide water balance:

- Table 18-2 shows the estimated effluent flows of the main mine components.
- Figure 18-6 shows the required capacity of the WTP on a monthly basis for the operation phase of the mine, considering the 95% non-exceedance probability (p95%) scenario. According to Figure 18-6, the required capacity of the WTP is 650 m^3/hr in the first four years of operations, increases to 750 m^3/hr in years 5 to 8, then increases to 962 m^3/hr (1,000 m^3/hr is considered in the design), and finally decreases in the last two years of the operations. This flow will be combined with the flow from the sewage treatment facility (3.6 m^3/hr) and discharged to Springpole Lake.

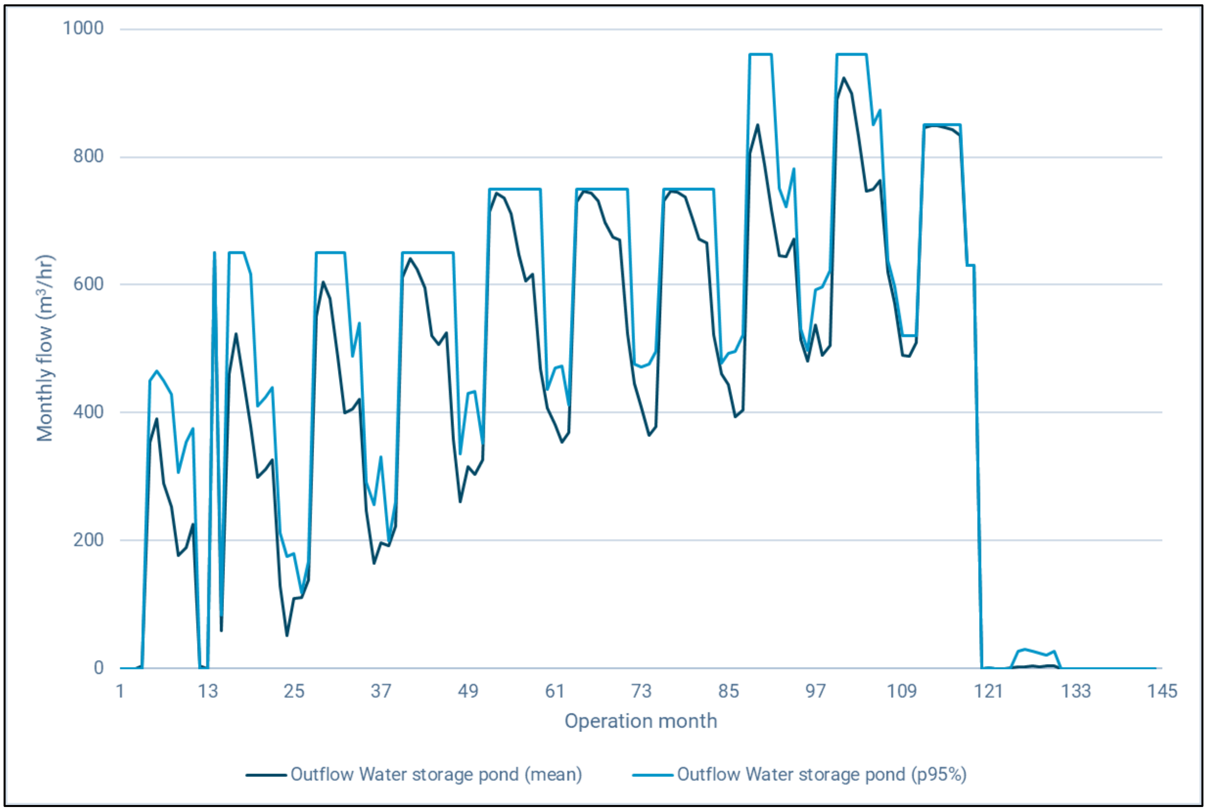
Table 18-2: Estimated Effluent Flows of the Main Mine Components

Facility	Year 1 to year 10 (m3/hr.)		Year 11 to year 12 (m3/hr.)	
	Monthly average	p95% (1)	Monthly average	p95% (1)
Open Pit	520	1,103	0.0	0.0
Waste Storage Facility Outflow to the Process Plant	113	150	123	150
Low Grade Ore Stockpile	29	159	33	151
Water Storage Pond Outflow to the Water Treatment Plant (2)	496	960	1	31
Water Storage Pond Outflow to the Process Plant (3)	55	168	41	166

Notes: 95% non-exceedance probability.

Make-up water for the process plant is supplied from two sources, primarily from the WSF and secondarily from the water storage pond.

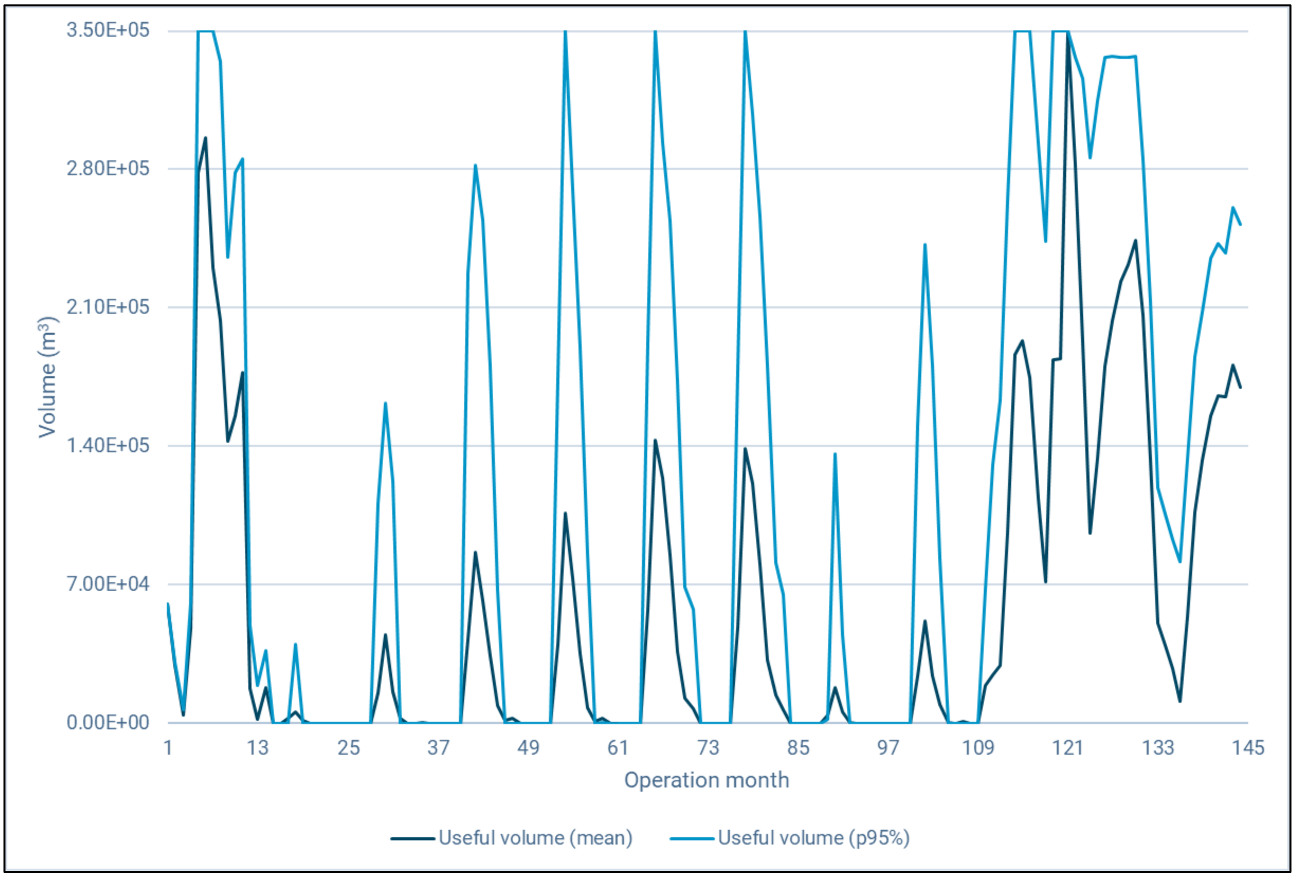
Figure 18-6: Water Storage Pond Outflow to the Water Treatment Plant



Source: AGP, 2021

- Figure 18-7 shows the required capacity of the water storage pond on a monthly basis for the operation phase of the mine. Considering the 95% non-exceedance probability (p95%) scenario, the maximum required volume for the pond is 350,000 m^3.
- Make-up water for the process plant is supplied from two sources, primarily from the WSF and secondarily from the WSP. The water balance model indicated that the required make-up water (168 m^3/hr) for the process plant is available during the entire operation phase of the mine (year 1 to year 12) except for a few months in year 12 (January, February, March, November, and December) due to zero outflow from the pit dewatering. During these months, the make-up water will be pumped to the process plant from the open pit.

Figure 18-7: Required Volume of the Water Storage Pond



Source: AGP, 2021

18.7 Built Infrastructure

The built infrastructure requirements are summarized in Table 18-3.

Table 18-3: Built Infrastructure Requirements

Item	Size	Comment
Process Buildings		
Primary Crusher Building	25.5m x 19.5m (497 m^2)	• Pre-engineered, steel frame metal cladded building • 85 t overhead crane, 17m span. Comes with dump pocket cover 19.5 m W x 18.5 m L
Grinding Building	48 m x 42 m (2,016 m^2)	• Pre-engineered, steel frame metal cladded building • 60 t sag mill O/H crane, 20m span • 45 t ball mill overhead crane, 20m span
Process Building	88 m x 42 m (3,696 m^2)	• Pre-engineered, steel frame metal cladded building • 25 t regrind mill overhead crane, 20m span • 25 t regrind / flotation over crane, 20m span
Stockpile Building	70 m x 70 m (4,900 m^2)	• Pre-engineered, fabric cladded dome building •
Gold Room	35 m x 14 m (490 m^2)	• Pre-engineered, steel frame metal cladded building
Tailings Filtration Building	130 m x 38m (4,940 m^2)	• Pre-engineered, steel frame metal cladded building
Reagent Building	70 m x 30 m (2,100 m^2)	• Pre-engineered, steel frame metal cladded building • 7.5 t overhead crane
Infrastructure Buildings		
Administration / Dry (Mine & Plant)	45 m x 16 m (720 m^2)	• Modular Building • Houses the site management team, including general management, commercial and administration management, engineering, senior processing, and maintenance personnel. Will be serviced with power, water, sewage, air conditioning and heating, communications.
Mine Offices	48 m x 18 m (864 m^2)	• Modular Building • Houses management and administration team for mining specific operations • Houses mine rescue room and equipment
Warehouse / Workshops (mine/plant)	36 m x 18 m (648 m^2)	• Pre-engineered, steel frame fabric cladded building • For maintenance for process equipment, as well as for the storage of equipment spare parts
Gatehouse	10 m x 4.2 m (42 m^2)	• Modular Building
Laboratory (including lab equipment)	18 m x 12 m (215 m^2)	• Modular Building • Includes sample receiving and preparation, fire assay, weighing room, wet analytical laboratory, dry instrument room, and utilities and storage modules. • Houses the laboratory equipment for assaying, metallurgical, and environmental requirements. Dust-collection equipment will be located external to the laboratory building. The building will be serviced with power, water, air conditioning and heating, communications, air and mercury scrubbers, and fume hoods.
Tire Change Shop	24 m x 20m (480 m^2)	• Pre-engineered, steel frame fabric cladded building • Primarily mine haul fleet tire maintenance

Item	Size	Comment
Truck Shop (mine & plant)	43 m x 45 m ($1935\ m^2$)	• Pre-engineered, steel frame metal cladded building • Two (2) 25 t overhead crane • Separate sections for warehousing spare parts and tool storage and the other for maintenance workshop
Fuel Depot	TBD	• ULS diesel for light and mine fleet for 3-5 days' supply (150,000 - 250,000 L) • Diesel exhaust fluid (DEF) storage (20,000L by totes), and dispensing • Other minor fuel types of storage
Vehicle wash-bay	10 m x 20 m ($200\ m^2$)	• Pre-engineered, steel frame fabric cladded building • Co-located with truck shop • Contains fluid-collection sump and oil-water separator that will be located adjacent to the truck workshop and warehouse. Wash water will be collected in sump where settling will occur and passed through oil-water separator prior to the water being recirculated back to the wash system.
Explosive Storage Magazine	TBD	• Will consist of a powder magazine in accordance with current applicable explosives regulations. Dirt berms will be placed around the magazines for additional security. Explosives will be delivered to site by vendors using the main access.

18.8 Camps and Accommodations

A 450 to 500-person construction camp is envisaged. This camp will be reduced substantially following construction for use as the permanent camp.

18.9 Power and Electrical

Average electrical demand site-wide is estimated to be 55 MW. The PFS envisaged that power supply to the proposed Springpole mine and process plant come from a connection to the provincial distribution grid. This would be by a new 75 km south-east ward, 230kV, single circuit power overhead transmission line which would provide tie-in to a new 230kV line between Dinorwic and Pickle Lake which is currently being constructed by Wataynikaneyap Power. As alternatives to the 230kV transmission lines, 115 kV and 138 kV have been considered, but the 230 kV option prevails in infrastructure cost and reduced power loss in the study conducted by Nordmin Engineering on behalf of First Mining.

The incoming electrical power from the 230 kV transmission line will be stepped down at the prefabricated substation to 25 kV for in-plant distribution through one 230/25 kV step-down transformer. All required auxiliary services, control room for substation operation, will be housed within the substation perimeter fence. The main substation control and automation system is designed for centralized operation of the substation, with a communication link to the plant-wide process control system (PCS).

The 25 kV distribution switchgear will be situated in the prefabricated main electrical room located next to the main substation. From the 25 kV switchgear, power will be supplied to all electrical rooms within the plant site through cable trays or via underground duct banks as needed. Overhead powerlines will feed distant facilities such as mine area, camp area, and water supply pumps.

There are total of six electrical rooms planned, including the main electrical room and regrind mill VFD room and those in various crushing and processing areas. All rooms will be prefabricated with internal electrical equipment pre-loaded prior to delivery to site.

Variable frequency drives have been allowed where required and will be fed from the main 25 kV switchgear location. All medium-voltage motors or drives will be fed from 4.16 kV switchgears, and starters for low-voltage motors will be grouped in motor control centers (MCC), with incoming breakers. The MCC's will be in the electrical room and will include intelligent combination starters, with circuit breakers for instantaneous fault protection.

19 MARKET STUDIES AND CONTRACTS

19.1 Markets

The gold markets are mature global markets with reputable smelters and refiners located throughout the world.

Gold is a principal metal traded at spot prices for immediate delivery. The market for gold trading typically spans 24 hours a day within multiple locations around the world (such as New York, London, Zurich, Sydney, Tokyo, Hong Kong, and Dubai). Daily prices are quoted on the New York spot market.

19.2 Gold Price

First Mining has not completed any formal marketing studies with regard to gold production that will result from the mining and processing of gold ore from the Project into doré bars. Gold production is expected to be sold on the spot market. Terms and conditions included as part of the sales contracts are expected to be typical of similar contracts for the sale of doré throughout the world. There are many markets in the world where gold is bought and sold, and it is not difficult to obtain a market price at any particular time. The gold market is very liquid with a large number of buyers and sellers active at any given time.

The Mineral Resources were estimated at a gold price of USD$1,550/oz and at a silver price of USD$20/oz. As of December 2020, the median consensus price forecast from 33 investment dealers estimated a gold price of USD$1,608/oz and a silver price of USD$20.54/oz long term. As of January 14, 2021, the trailing two-year gold price was USD$1,595/oz and silver price was USD$18.57/oz and the trailing three-year gold price was USD$1,486/oz and silver price was USD$17.60/oz.

For the purpose of the 2021 Pre-Feasibility Study, a gold price of USD$1,600/oz and silver price of USD$20/oz were assumed. The exchange rate used in the study is CDN$1.00:USD$0.75.

19.3 Contracts

The mine plan assumes that doré will be shipped from site to major refineries. First Mining will enter into a refining agreement with various refiners around the world when the timing is appropriate. The terms and conditions will be consistent with standard industry practices. Refining charges include treatment and transportation.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Setting

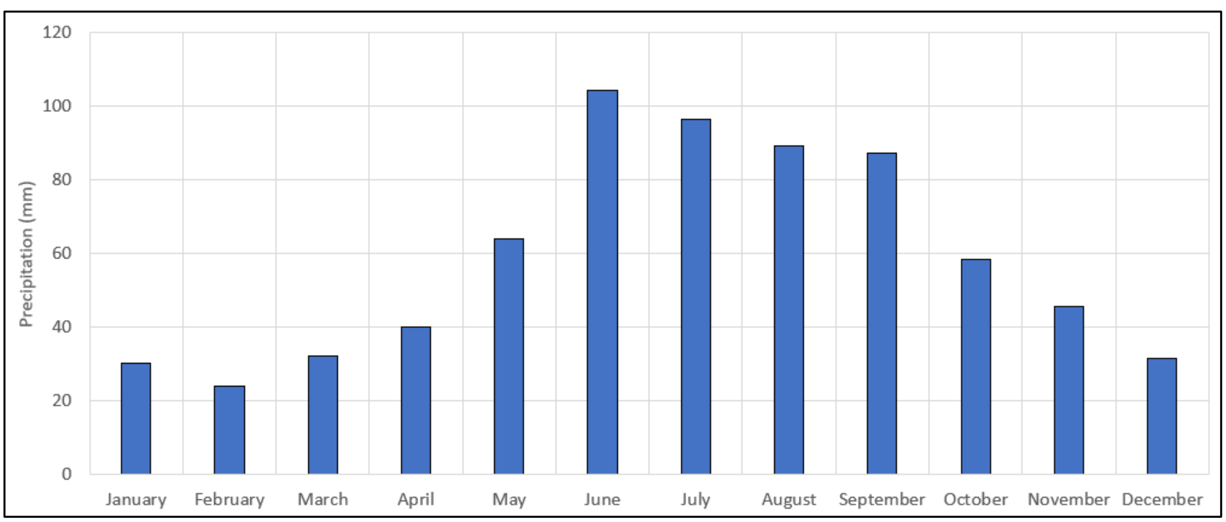
The Project site is in a remote area of northwestern Ontario. First Mining, and its predecessor Gold Canyon, have been collecting environmental baseline data to support the Project's EA since 2010, and data collection is ongoing. These studies are primarily focused on characterizing biological and physical components of the aquatic and terrestrial environments that may be impacted by and may interact with the proposed Project. The dataset compiled to date within these programs exceeds the level of environmental baseline data one would typically have in support of a PFS.

20.1.1 Biophysical Setting

Climate

The Project meteorological station was originally installed in September 2011. Several instrumentation updates were made to the station over the years, including the addition of an evaporation pan. The PEA Update (SRK, 2019) included an assessment of relevant climate parameters including air temperature, precipitation, and evaporation. These estimates were based on the Canadian Climate Normals 1971-2000 (Environment Canada, 2012). Average daily air temperatures ranged between -40°C and 0°C in January, and between 20°C and 40°C in July. Precipitation data were analyzed using the Canadian Climate Normals for nine stations surrounding the Project. The mean annual precipitation was determined to be 704 mm, and Figure 20-1 shows the monthly precipitation distribution. Probabilistic extreme rainfall estimates were made using the Rainfall Atlas of Canada (Hogg and Carr, 1985).

Figure 20-1. Average Monthly Precipitation Distribution



Source: Swiftwater, 2020

Lake evaporation was calculated with evaporation estimating software using Canadian Climate Normal data from five stations near to the Project. The average annual evaporation is estimated to be 546 mm (SRK, 2019).

Air Quality & Noise

There are no historical records of air quality or noise in the area immediately surrounding the Project, in large part because there are no industrial noise or air emission sources near to the Project. Potential nearby sources of noise and air emissions include forest fires, combustion products from heating oil and propane that are used for residential and recreational purposes at the numerous tourist lodges, and periodic timber harvesting activities in the Trout Lake Forest.

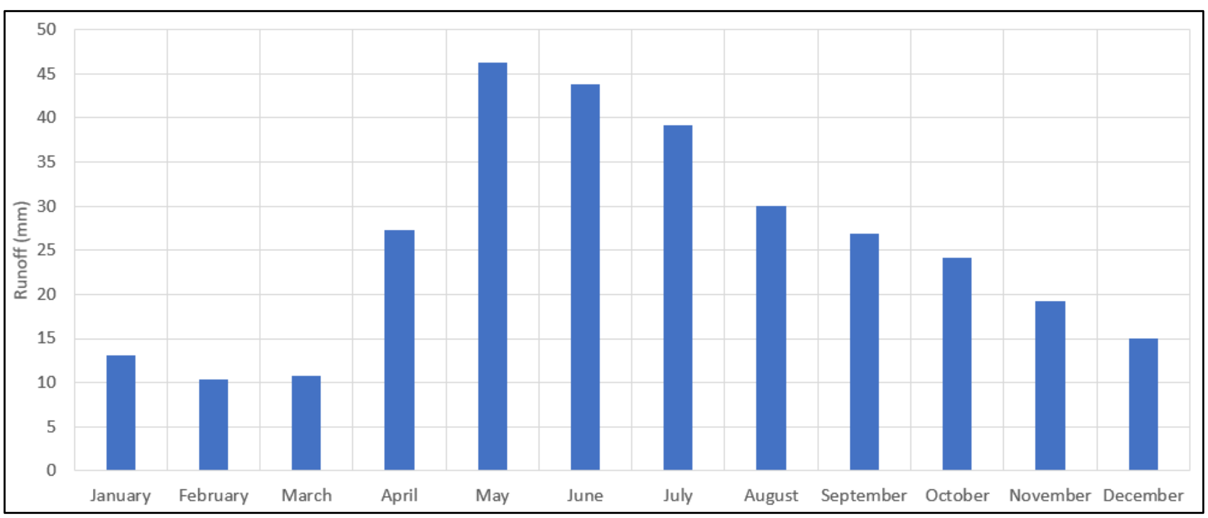
Terrain and Soils

The property is overlain by glaciated terrain characteristics common throughout the Canadian Shield. The land is generally of low relief with less than 30 m of local elevation change, and typically separated by a series of interconnected shallow ponds and lakes (SRK, 2019). Surficial geology consists of silty and sandy till blanket to veneer less than 4 m (Pinchin, 2014). Bedrock outcrops are limited and are generally covered by moss or muskeg. Soils are predominately grey wooded and podsols on well-drained sites, and peat and gleysols are predominant in poorly drained areas.

Streamflow

Historical daily streamflow rates were obtained from Environment Canada (2010) for five unregulated gauging stations surrounding the Project. The daily flow rates were used to determine annual run-off, monthly flow rates and monthly distribution. The average of the values from the five stations was used to estimate site-specific values. The average annual run-off was estimated to be 306 mm. The monthly run-off distribution is included in Figure 20-2.

Figure 20-2: Average Monthly Run-off Distribution



Source: Swiftwater, 2020

Five hydrometric stations were installed in June 2011 and consist of a pressure transducer mounted inside a protective slotted PVC pipe, recording water level every 15 minutes. A staff gauge is mounted at each site (SRK, 2019). Flow measurements were conducted during site visits with a flow meter. The highest flows on site were measured in the freshet, and the lowest flows in the late summer and fall before freeze-up.

Groundwater

Hydrogeological data collection began in 2013 and included 20 packer tests in seven core holes drilled within the proposed pit footprint. A full interpretation of the data was not available for the previous PEA reporting. Additional groundwater investigations were carried out in 2017 and 2018 (North Rock, 2018, 2019), to provide a groundwater monitoring system across the site and initiate long-term site-wide data collection. Furthermore, monitoring wells were installed in five overburden sites (MW1 to MW5) and in six deeper exploration boreholes (DDH1, DDH2, BL-235, BL08-235, SP11-102, and SP11-064) located across the study area. Groundwater levels and samples were obtained from these wells and form the basis of a baseline hydrogeochemical database. Experience in this environment suggests that groundwater flow will mostly follow surface water flow directions.

Fish & Fish Habitat

Springpole Lake has a predominantly rocky shoreline and contains numerous islands and rocky shoals. The Birch River is its largest tributary, and it enters at the southwest end of Springpole Lake through a short section of rapids downstream from Cromarty Lake. There are also several small tributary streams flowing into Springpole Lake. The outflow of Springpole Lake is also through the Birch River, at the east end, into Gull Lake. Springpole Lake has a surface area of 2,861 ha; the Project footprint includes 6% of that area. Overall, Springpole Lake has a maximum depth of approximately 40 m and an average depth of 6.3 m. The portion of Springpole Lake that is within the Project footprint has an average depth of 13 m.

The northern portion of Springpole Lake, within which the Project is located, is 4.5 km wide and 6.5 km long. It is generally deeper and more open than the east arm of the lake and has three large basins exceeding 30 m in depth. The lake has three deep additional basins exceeding 20 m in depth. The east arm of Springpole Lake is 17.7 km long. Much of that length is a narrow channel bound by a steep bedrock wall along the north shore.

Beginning in 2011, an extensive and comprehensive assessment program was carried out to describe the fish community and habitat in surface water bodies that are within the Project area. Fish and fish habitat studies have been carried out on Springpole Lake, 12 small unnamed waterbodies, and 9 small tributary streams in the Project area.

Springpole Lake as well as nearby Birch and Seagrave Lakes support diverse fish communities including sport species common to both cold-water (lake trout and whitefish) and cool-water lakes (walleye, northern pike, and yellow perch), and a number of other non-game and forage fish. Among the twelve small lakes surveyed; six host fish communities that include sport species including yellow perch and northern pike; four host only forage species; and two are considered devoid of fish. Of the nine small streams surveyed, the largest and best connected to lakes support yellow perch and other forage fish. The smaller, more ephemeral streams typically support only forage fish species.

Lake trout are utilizing multiple deep basins within Springpole lake as cold-water refuges during the summer months and move freely to both upstream and downstream lakes in the fall and return in the late spring. There are several lake trout spawning shoals in Springpole Lake. No walleye have been observed spawning within the Project footprint in Springpole Lake, and no lake sturgeon or other endangered fish species are present in Springpole Lake.

Vegetation and Ecological Communities

The Project is part of the Lac Seul Upland, which extends eastward from Lake Winnipeg in Manitoba to the Albany River in northwestern Ontario. Forest composition on the Project is typical of the Lac Seul Upland. Dominant tree species include trembling aspen, black spruce, white birch, balsam fir, and white spruce and jack pine. Understory ground cover species composition and abundance is typical of mesic mixed wood boreal sites and lacks microhabitats likely to harbor rare vascular plant species. A variety of common, early successional graminoids and herbaceous ground cover plants are prevalent on areas of the Project where mature timber has been removed or where the canopy is open and exposed to direct sunlight. Natural re-vegetation and succession has been observed to be rapid in areas of historical exploration. Ecological Land Classification communities present at the Project site include deciduous forests (FOD), mixed-wood forests (FOM), coniferous forests (FOC), deciduous swamp (SWD), mixed-wood swamps (SWM), and coniferous swamps (SWC).

A field campaign was undertaken in 2012 to evaluate vegetation in the vicinity of the Project site. Provincial Ecosite B049 is the most common ecosite in the forest (38 % of stands). Riparian sites assessed within the study area are dominated by shrubs and had the maximum species richness, upland areas had the maximum species evenness. None of the provincially significant species listed in the NHIC database were encountered during the field surveys (First Mining Gold Corp., Project Description, 2018).

Wildlife and Wildlife Habitat

Desktop and field baseline work has been ongoing in the Project area since 2011. Wildlife habitat modelling for selected species was completed using the Ontario Landscape Tool (OLT) for those portions of the study area that had forest resource inventory (FRI) data. The OLT uses these data to develop prescriptive indicators such as conifer age class distribution and landscape cover, as well as evaluative indicators of wildlife habitat quality and distribution. These evaluative indicators include the spatial identification of wildlife habitat for seven focal wildlife species such as woodland caribou, moose, marten, lynx, snowshoe hare, fisher, and northern flying squirrel, as well as many species of songbirds.

Field work was undertaken in 2011-2012, with a supplemental ungulate aerial survey in winter 2013. Species at risk (SAR) identified in the study area include indications of the presence of wolverines, northern myotis/little brown myotis, and woodland caribou (MNRF, 2009b). The only colonial nesting birds located within the study area were Bonaparte's gulls, though potential nest sites for ring-necked duck are not considered rare within the study area (there are grassy sites within 200 m of water: MNRF, 2010). There is no evidence of nest fidelity in ring-necked ducks, nor does this site support large concentrations of nesting waterfowl, other species of conservation concern, or a variety of waterfowl species. There are several moose calving sites located within the study area, but outside of the Project site. A mineral lick has been identified on an island in Springpole Lake south of the Project site, but there are no known mink, otter, or fisher denning sites within the study area. No habitat of the provincially rare species listed was identified during field investigations (DST, 2013).

20.1.2 Social and Community Setting

Human Environment

The Red Lake area has been a historic mining district since the gold rush of the 1920s, and it currently has five active mining projects and other decommissioned/abandoned mines situated within the municipality of Red Lake. The mining and mineral development sector is the largest employer in the region. The region hosts remote tourism operations and seasonal camps, including on Birch Lake which is situated upstream of the Project site. Other remote tourism lakes in the general vicinity of the Project site include Seagrave, Bertha, Deaddog, Gull, Fawcett, and Christina. Other proximal seasonal residences are located south of the Project, on Johnson Island. The Project is within the Trout Lake Forest and forestry activities are ongoing in the region, in accordance with the Crown Forest Sustainability Act.

The Project is located approximately 145 km north of the Municipality Sioux Lookout. The Municipality of Sioux Lookout is a progressive community that supports respectful and sustainable resource development. The Municipality of Sioux Lookout is often referred to as the Hub of the North, connecting remote northern communities to healthcare and essential services. The Municipality has expressed interest in establishing a road connection to the Project's access road to service the mine and First Mining will continue to engage and support the Municipality in this regard.

Cultural Heritage

Archeological assessment work led by a licensed professional archaeologist has identified archaeological sites at the Project site. Large setbacks from these sites will be maintained in the base case general arrangement. Ongoing engagement with the Indigenous community representatives will help to identify additional sites and mitigation strategy as required.

Indigenous Communities

The Project is approximately 40 km from Cat Lake First Nation, 45 km from Slate Falls First Nation, and 120 km from Lac Seul First Nation.

First Mining is working with the local Indigenous communities it is in consultation with to initiate and continue Traditional Knowledge and Land Use (TK/TLU) studies for the Project area. It is understood that communities hunt, trap, fish, and collect a variety of species, including:

- Wildlife: moose, deer, mink, muskrat, rabbit, otter, beaver, fox, bobcat, weasel, squirrel, wolf, and marten
- Birds: partridge and ducks
- Fish: sturgeon, walleye (also referred to as pickerel), northern pike, whitefish, and trout
- Plants: wild rice, blueberries, raspberries, cherries, juniper, sage, sweet grass, willow, cedar, and tree barks

First Mining intends to incorporate the findings from the TK/TLU studies completed by the local Indigenous communities in the Project area to further scope and refine the Project moving forward.

20.2 Environmental Management

20.2.1 Site Management and Monitoring

This section describes the general approach to protect the environment and minimize potential impacts of the Project.

Environmental Design Basis

Preliminary environmental design criteria have been developed for Project features that have the potential to release contaminants to air, water, and land. Strategies included in the environmental design basis include, but are not necessarily limited to:

20.2.1..1 Emissions to Air

- strategic positioning and abatement for noise sources that include, but are not necessarily limited to fans, compressors, generators, crushers, pneumatic transfer systems and mobile equipment
- strategic positioning and abatement for point sources and non-point sources of air emissions
- strategic positioning of light sources to minimize light contamination

20.2.1..2 Emissions to Water

- mine and process water management system and associated management system for residuals from the water treatment process
- domestic sewage management systems for black water and grey water
- management system for wastewater from equipment wash bays, surface shops and service buildings
- management system for run-off from areas of site where precipitation and snowmelt have contacted ore and/or waste rock
- management system for run-off from areas of site where precipitation has not contacted ore and/or waste rock

20.2.1..3 Emissions to Land

- ore management system including provisions to minimize spillage and dispersion
- waste rock management system, including provisions to minimize spillage and dispersion
- solid and liquid waste management system
- petroleum products management system
- spill prevention and response system

In addition to above noted engineered Project features, First Mining will develop monitoring programs and proactive adaptive management plans to mitigate environmental impacts that are identified by the monitoring program.

Environmental, Health, and Safety Management System

First Mining will develop an environmental, health and safety ("EHS") management system to address the EHS needs of the Project based on the results of the Environmental Impact Statement, which will include a range of plans and programs.

20.2.2 Waste Management – Waste Rock and Tailings Disposal

Approximately 117 Mm^3 of mine tailings and PAG waste will be stored within the WSF, including 76 Mm^3 of tailings and 41 Mm^3 of PAG waste rock. The construction of perimeter embankment dams will improve stability and provide freeboard for run-off collection as well as storage for storm events.

20.2.3 Water Management

The surface of the filtered tailings and PAG waste rock within the WSF will be graded to encourage flow to defined sumps/pumping points. The collected surface water will be directed to a contact water management pond (CWMP) located southeast of the WSF. The water stored in the CWMP will be used to supplement mineral processing and/or treated and is currently planned to be released to Springpole Lake.

Water that infiltrates the surface will fill any void space remaining within the waste materials and may end up contributing to elevation of the phreatic surface at the base of the facility (the phreatic surface may also be elevated because of loading from subsequent waste placement if the material is placed at or near saturation). A drainage system will include pumping stations (drainage ponds) with submersible pumps installed within screened HDPE pipe to facilitate collection of water and transfer to the CWMP. The HDPE pipe will be extended as the waste surface rises. The primary drainage ponds and pumping systems will be installed at topographical low points within the WSF footprint.

20.3 Permitting

20.3.1 Federal Environmental Assessment

A Project description was submitted to the Canadian Environmental Assessment Agency in February 2018, with EIS Guidelines being issued by the Canadian Environmental Assessment Agency in June 2018. Issuance of the EIS Guidelines confirmed that the Project is subject to environmental assessment under *CEAA 2012*, and associated regulations designating physical activities rather than under the new federal Impact Assessment Act (IAA).

20.3.2 Federal Permitting Requirements

In addition to the requirement for assessment under CEAA, 2012, key federal permits that may be required pending further regulatory advice:

- Fisheries Act Authorization (Fisheries and Oceans Canada (DFO))
- Canadian Navigable Waters Act (Transport Canada)
- Schedule 2 of Metal and Diamond Mining Effluent Regulations (MDMER)

Prohibitions under other pieces of federal legislation also apply but no permitting requirements are currently expected. These may include, but would not necessarily be limited to, the following:

- Canadian Environmental Protection Act, SC 1999
- Migratory Birds Convention Act, SC 1994, c22
- Explosives Act, RSC 1985, C. E-17
- Transportation of Dangerous Goods Act, SC 1992, c. 34
- Species at Risk Act, SC 2002; c. 29
- Nuclear Safety Control Act, SC 1997, c. 9)

20.3.3 Provincial Environmental Assessment

First Mining has entered into a Voluntary Agreement with the Ministry of Environment, Conservation and Parks (MECP), formally Ontario Ministry of the Environment and Climate Change, to undertake an Individual EA, under Section 3.0.1 of the provincial *Environmental Assessment Act*.

20.3.4 Provincial Permitting Requirements

Based on the current understanding of the Project area and Project description provided by First Mining, it is expected that the following permits and approvals will be required:

- Mine Closure Plan, Mining Act, Energy, Northern Development and Mines
- Permit to Take Water, Ontario Water Resources Act, MECP
- Environmental Compliance Approval (Air/Noise), Environmental Protection Act, MECP
- Environmental Compliance Approval (Sewage), Ontario Water Resources Act, MECP
- Environmental Compliance Approval (Waste), Environmental Protection Act, MECP
- Work Permit, Public Lands Act, Ministry of Natural Resources and Forestry (MNRF)
- Work Permit, Lakes and Rivers Improvement Act, Ministry of Natural Resources and Forestry (MNRF)
- Aggregate Permit, Aggregate Resource Act, MNRF
- Overall Benefit Permit, Endangered Species Act, MECP
- Forestry Resource Licence/Release of Reservation, Crown Forest Sustainability Act, MNRF
- Archaeological Clearance, Ontario Heritage Act, Ministry of Heritage, Sports, Tourism, and Culture Industries (MHSTCI)

20.4 Engagement and Consultation

20.4.1 Indigenous Communities

The federal government identified Cat Lake First Nation, Slate Falls First Nation, Lac Seul First Nation, Wabauskang First Nation, Mishkeegogoamang Ojibway Nation, Ojibway Nation of Saugeen, and Métis Nation of Ontario in 2018 (updated in 2020), while in 2018 the provincial government identified Cat Lake First Nation, Slate Falls First Nation, Lac Seul First Nation, Wabauskang First Nation, Mishkeegogoamang Ojibway Nation, Ojibway Nation of Saugeen, Pikangikum First Nation, and Métis Nation of Ontario, as potentially impacted by the Project or having an interest in the Project.

In March 2017, the First Nations of Cat Lake, Slate Falls and Lac Seul entered into a Shared Territory Protocol Agreement. These three First Nations are known collectively as the Shared Territory Protocol Nations ("STPN"). In February 2018, First Mining entered into a Negotiation Protocol Agreement with the STPN and will continue information sharing and consultation throughout the EA process.

20.4.2 Public Engagement

Potentially affected stakeholders have been identified based on previous permitting and EA processes for work on the Project. Stakeholders include remote tourism operators, seasonal residences, forestry interests, local municipalities, and education/training organizations.

20.4.3 Government

First Mining has been engaging with federal and provincial agencies on the Project with the development of the Project description. Correspondence includes in-person meetings, emails, and phone calls with:

- Impact Assessment Agency of Canada
- Ministry of Energy, Northern Development and Mines
- Ministry of the Environment, Conservation and Parks
- Ministry of Natural Resources and Forestry
- Fisheries and Oceans Canada

20.5 Mine Closure and Rehabilitation

A general rehabilitation and closure approach to meet the objectives of Ontario Mines Act and Regulation 240/00 has been developed and described below. The overall objective for closure is to return the Project site to a productive condition after mining is complete that is capable of supporting plant, wildlife and fish communities, and other applicable land uses.

20.5.1 Rehabilitation Planning

20.5.1..1 Operation

Progressive Rehabilitation takes place during operations in a phased approach as facilities are developed.

20.5.1..2 Closure

Closure decommissioning and rehabilitation takes place once mining has permanently stopped. Activities will include decommissioning of infrastructure and buildings, site preparation and planting of disturbed sites, and surface preparation to facilitate natural revegetation. Closure is expected to be completed three years after operation ceases.

20.5.1..3 Post-Closure

Post closure activities typically include site security, water treatment, and environmental/reclamation monitoring to ensure progression towards closure objectives. Post closure monitoring and assessments will continue until the site has met the rehabilitation objectives at which time the property would be released to the Crown.

20.5.2 Proposed Approach to Closure and Rehabilitation

Closure concepts will be refined as engineering and modelling advances during the EA and permitting. Risks will also be better quantified resulting in a more robust set of site closure objectives and a more detailed closure plan. The current expectation includes, but is not limited to:

- During construction of the Project, land clearing should be limited to areas necessary to complete the work.
- Organic soil, overburden, and glacial till will be salvaged and stockpiled.
- Opportunities for progressive rehabilitation will be reviewed and implemented during operations, as is reasonable, to minimize erosion and manage invasive species.
- The overall objective of closure is to return the Project site to a productive condition after mining is complete that is capable of supporting plant, wildlife and fish communities, and other applicable land uses, as stated above.

20.5.3 Progressive Rehabilitation

Some potential opportunities for progressive rehabilitation may include:

- Decommissioning and salvage of structures used solely for exploration and construction. Road and pad revegetation to reduce erosion and manage invasive species.
- Reclamation research studies and trials are budgeted for years 8 to 10 of operations and not included in closure costing.
- Early rehabilitation of the WSF embankments:
  - The WSF embankment will be constructed to the maximum footprint/elevation in year six of operation.

20.5.4 Closure Rehabilitation

The closure activities associated with the major components of the Project are described in the following sections. The general closure activities that will be completed for the site will include:

- pit lake development
- removal of hazardous substances
- cleaning and removal of equipment
- demolishing and salvage of site buildings and infrastructure
- infilling or breaching of water storage ponds
- water treatment
- ground cover application, re-sloping, and revegetation of disturbed sites

21 CAPITAL AND OPERATING COSTS

21.1 Summary

The following basic information pertains to the estimate:

- cost estimate base date is Q4, 2020
- expressed in Canadian dollars (CDN) unless otherwise noted
- currency exchange rate 0.75 USD: 1 CDN

The cost estimate is based on an engineering, procurement, and construction management (EPCM) implementation approach. The capital cost estimate has an accuracy of -20% / +30% (AACE Class 4). The estimate includes the cost to complete the design, procurement, construction, and commissioning of all the identified facilities.

The estimate was derived from a number of fundamental assumptions as indicated in process flow diagrams, general arrangements, mechanical equipment list, electrical equipment list, material take-offs (MTOs), electrical layouts, scope definition and a work breakdown structure. The estimate included all associated infrastructure as defined by the scope of work.

The study was completed in Canadian dollars (CDN) and is reported in United States dollars (USD) for easy comparison to other projects nominated in that currency.

The initial and life of mine capital cost estimate is summarized in Table 21-1. The operating cost estimate is shown in Table 21-2.

Table 21-1: Springpole Capital Cost Estimate (USD)

Cost Type	Cost Description	Project Capital ($USD M)		
		Initial	Sustaining	Total
Direct	Mine	144.5	51.3	195.8
	Site Development	21.0	-	21.0
	Process Plant	296.7	4.2	300.9
	On-site Infrastructure	38.4	-	38.4
	Off-site Infrastructure	35.3	-	35.3
	Direct Subtotal	**535.9**	**55.5**	**591.4**
Indirect	Indirects	47.9	-	47.9
	EPCM Services	37.5	-	37.5
	Owner's Costs	16.1	-	16.1
	Indirect Subtotal	**101.4**	**-**	**101.4**
Provisional	Contingency and Management Reserve	80.9	-	80.9
Closure	Closure Costs	-	29.5	29.5
Total		**718.3**	**85.0**	**803.3**

Table 21-2: Springpole Operating Cost Estimate (USD)

	Units	Life of Mine (Year 1-12)
Open Pit Mining	$/t moved	1.46
	$/t ore	6.52
Processing	$/t ore	10.87
G&A	$/t ore	0.79
Total Operating Cost	**$/t ore**	**18.18**

The Canadian cost estimate for capital and operating costs are also tabulated and shown in Table 21-3 and Table 21-4.

Table 21-3: Springpole Capital Cost Estimate (CDN)

Cost Type	Cost Description	Project Capital (CDN$ M)		
		Initial	Sustaining	Total
Direct	Mine	192.6	68.4	261.0
	Site Development	28.0	-	28.0
	Process Plant	395.7	5.5	401.2
	On-site Infrastructure	51.2	-	51.2
	Off-site Infrastructure	47.1	-	47.1
	Direct Subtotal	**714.6**	**73.9**	**788.5**
Indirect	Indirects	63.8	-	63.8
	EPCM Services	50.0	-	50.0
	Owner's Costs	21.4	-	21.4
	Indirect Subtotal	**135.2**	**-**	**135.2**
Provisional	Contingency and Management Reserve	107.9	-	107.9
Closure	Closure Costs	-	39.3	39.3
Total		**957.7**	**113.3**	**1,071.0**

Table 21-4: Springpole Operating Cost Estimate (CDN)

	Units	Life of Mine (Year 1-12)
Open Pit Mining	$/t moved	1.94
	$/t ore	8.69
Processing	$/t ore	14.50
G&A	$/t ore	1.06
Total Operating Cost	**$/t ore**	**24.24**

21.2 Capital Cost Estimate

21.2.1 Capital Cost Estimation

The facilities will consist of an open pit mine and process plant divided into the following areas: i) primary crusher ii) SAG and ball mills iii) flotation iv) tails leaching and CIP v) concentrate regrind/leaching/CIP vi) ADR plant vii) electrowinning and refinery viii) tails thickening/cyanide destruction ix) reagent mixing and x) dry-stack tailings management as well as associated infrastructure.

The estimate includes the cost to complete the design, procurement, construction, and commissioning of all the identified facilities.

The physical facilities and utilities for the Project include, but are not limited to, the following areas in Table 21-5 as noted by work breakdown structure (WBS) areas and their associated contributors.

Table 21-5: Springpole Work Breakdown Structure

WBS	Description	Capital Cost Contributor
1000	**Mining**	
1100	Mining Development Surface	AGP
1200	Mine / Bay Dewatering	AGP (Mine), Knight Piésold (Lake)
1300	Haul Roads	AGP
1400	Waste Rock Storage	AGP & Knight Piésold
1500	Ore Stockpile	SRK
1600	Mining Equipment	AGP
1700	Mine Ancillary Services	SRK
1800	Mine Explosives Magazine	AGP
2000	**Site Development**	
2100	Cofferdam A	Knight Piésold, AGP
2200	Cofferdam B	Knight Piésold, AGP
2400	Bulk Earthworks	SRK
3000	**Process Plant**	
3100	Crushing, Stockpile/Reclaim	SRK
3200	Grinding	SRK
3300	Flotation	SRK
3400	Flotation Tails Leaching	SRK
3500	Concentrate Leaching	SRK
3600	ADR Plant	SRK
3700	Refinery	SRK
3800	Reagents and Services	SRK
3900	Tailings Filtration	SRK
4000	**On-Site Infrastructure**	
4100	Power Supply and Distribution	SRK
4200	Ancillary Buildings	SRK
4300	Site Services	SRK
4400	Mobile Equipment	SRK
5000	**Off Site Infrastructure**	
5100	Off-site Roads	Existing – no work required
5200	HV Power Line (OHPL)	First Mining
5300	Water Supply	SRK
6000	**Project Indirects (Field, Camp, Vendors, Spares)**	**SRK, AGP**
7000	**EPCM**	**SRK, AGP**
8000	**Owner's costs**	**AGP, First Mining**
9000	**Provisions (Contingency)**	**AGP, SRK**

Estimate Structure

The estimate is broken out into direct costs and indirect costs of initial costs including sustaining capital and mine closure costs.

Direct costs are those costs that pertain to the permanent equipment, materials and labour associated with the physical construction of the process facility, infrastructure, utilities, buildings, etc. Construction contractor's indirect costs are contained within each discipline's all-in rates and considered as direct costs.

Indirect costs include all costs associated with implementation of the plant and incurred by the Owner, engineer or consultants in the design, procurement, construction, and commissioning of the Project.

Initial capital is the capital expenditure required to start up a business or in this case complete the construction of a facility to a standard where it is ready for initial production.

Sustaining capital is the capital cost associated with the periodic addition of new plant equipment or services that are required to maintain production and operations at their existing levels.

Basis of Estimate

The capital cost is a quantitative based cost estimate, with engineering developed MTOs with factored quantities, semi-detailed unit costs and budgetary quotations for major equipment.

The structure of the estimate is a build-up of the direct and indirect cost of the current quantities; this includes the installation/construction hours, unit labour rates and contractor distributable costs, bulk and miscellaneous material and equipment costs, any subcontractor costs, freight, and growth.

The methodology applied to develop the estimate is as follows:

- defined the scope of work
- quantified the work in accordance with standard commodities
- organized the estimate structure in accordance with agreed Work Breakdown Structure
- developed a priced Mechanical Equipment List and Electrical Equipment List
- determined bulk material pricing
- determined the installation cost for equipment and bulks
- established requirements for freight
- determined and agreed on foreign exchange rates
- determined growth allowances for each estimate line item
- determined/developed Indirect costs
- determined the estimate contingency value.
- undertook internal peer reviews and finalized the estimate.

The estimate source data included:

- equipment lists
- scope of work
- design criteria
- general arrangement drawings
- drawings and sketches
- structural models

- geotechnical investigation data
- process flow diagrams
- material take-offs
- equipment and bulks pricing
- contractor installation data
- vendor material supply costs
- historical data
- schedule

Various reference documents were used in the preparation of the estimate including:

- site topography
- historical and recent metallurgical test reports
- 2019 PEA
- power supply design
- Springpole PFS Implementation Plan

The pricing and delivery information for select major equipment, material and services was provided by suppliers based on the market conditions and expectations applicable at the time of developing the estimate. Limitations and risks to these quotations include market conditions and supply and availability during the actual execution phase.

The estimate base date is Q4 2020. The estimate was completed in CDN and converted into USD for various presentations.

Metric units of measure are used throughout the estimate.

Exchange Rates

The exchange rates used for converting foreign currency in the estimate are shown in Table 21-6.

Table 21-6: Estimate Exchange Rates

Code	Currency	Exchange Rate
CDN	Canadian Dollar	1.00 CDN = 1.000 CDN
USD	US Dollar	1.00 CDN = 0.750 USD
AUD	Australian Dollar	1.00 CDN = 1.045 AUD
EUR	Euro	1.00 CDN = 0.643 EUR

21.2.2 Mining Capital Cost

The mining capital cost estimate is grouped into various categories of the WBS. These include:

- 1100 - Mine Development - Surface
- 1200 - Mine – Bay Dewatering
- 1400 - Waste Rock Storage

- 1600 - Mining Equipment

The cost breakdown summary has been shown in Table 21-7.

Table 21-7: Mining Capital Cost Estimate

Mining Capital Category	Initial Cost (CDN$ M)	Sustaining Cost (CDN$ M)	Total Capital Cost (CDN$ M)
1100 - Mine Development - Surface	105.8	1.1	106.9
1200 - Mine – Bay Dewatering	5.2	2.6	7.8
1400 - Waste Rock Storage	42.9	43.3	86.2
1500 – Ore Stockpile	3.9	-	3.9
1600 - Mining Equipment	31.1	21.4	52.5
1700 – Ancillary Services	3.7	-	3.7
TOTAL	**192.6**	**68.4**	**261.0**

Mine Development – Surface (1100)

Mining activity commences in advance of the process plant achieving commercial production. This includes the movement of 19.6 Mt of waste and placement of 4.5 Mt of mill feed in a stockpile adjacent to the primary crusher. The mine operating costs associated with this time period are included in the capital cost estimate and expected to cost CDN$105.4 M. This cost covers all associated management, drilling, blasting, loading, hauling, support, engineering and geology departments labour, grade control costs and financing costs.

The mining during this time includes the development of a quarry for use in haul road construction and overall site requirements. A smaller sized equipment fleet (91 t trucks) will be used in road, cofferdam, tailings/waste storage facility preparation, and initial phase development.

The larger mining fleet will become operational in Year -1, when pre-production mining commences in earnest as the water level drops from bay dewatering. This work will widen/improve haul roads that had been pioneered, advancement of the WSF and continued stockpiling of mill feed in preparation of the process plant commissioning.

Pit electrification is included in this portion of the cost with some in Year -1 and extension of the electrical system in Year 1 as part of sustaining capital.

Mine – Bay Dewatering (1200)

The initial capital dewatering costs are primarily the bay dewatering behind the cofferdams. The total initial capital is CDN$5.2 M with CDN$4.7 M for dewatering of the bay.

Sustaining capital is the mine pit dewatering and this totals $2.6 M over the life of the mine.

Waste Rock Storage (1400)

The bulk of the cost is the associated earthworks at CDN$39.3 M.

The development of the waste rock storage facility by the smaller equipment fleet is included in the mine development cost (1100), but the cost associated with clearing and grubbing is included in this category. A total of 415 ha is cleared and grubbed with a unit cost of approximately CDN$8,000/ha for a total of CDN$3.6 M.

Ore Stockpile (1500)

Preparation of the ore stockpile locations which includes the foundation and drainage is covered in this cost category. This includes the high, medium, and low grade stockpiles adjacent to the plant and ROM hopper. The cost is only in the initial capital and totals CDN$3.9 M.

Mining Equipment (1600)

The mining equipment capital costs reflect the use of financing of the major equipment and most support equipment. Equipment prices used current quotations from local vendors. A 20% down payment is included in the capital cost for those units financed. The remaining cost is included in operating costs discussed later in Section 21-3. The cost for the base fleet management system to properly track mill feed, and waste types is included in this category.

Initial capital cost requirements totaled CDN$31.1 M with sustaining (new and replacement equipment) totaling CDN$21.4 M.

The base costs provided by the vendors are included in a calculation for each unit cost calculation and options added to that including the equipment fleet management modules. In the case of the electric/hydraulic shovels it also includes switchgear and initial trailing cable requirements. The capital cost if it was to be purchased outright is shown for comparison. The cost of financing and down payment of some of the major equipment is shown in Table 21-8 as used in the PFS.

Table 21-8: Major Mine Equipment – Capital Cost, Full Finance Cost and Down Payment

Equipment	Unit	Capacity	Full Finance Cost (CDN$)	Down Payment (CDN$)	Capital Cost (CDN$)
Production Drill	mm	140	1,174,000	229,000	1,115,000
Production Drill (Electric)	mm	251	5,591,000	1,061,000	5,307,000
Production Loader	m^3	23	8,190,000	1,555,000	7,773,000
Hydraulic Shovel (Electric)	m^3	36	17,703,000	3,360,000	16,803,000
Haulage Truck	t	240	5,701,000	1,082,000	5,411,000
Haulage Truck	t	91	1,971,000	374,000	1,871,000
Crusher Loader	m^3	13	3,066,000	582,000	2,910,000
Production/Support Excavator	m^3	7	2,33,000	424,000	2,119,000
Track Dozer	kW	474	2,391,000	454,000	2,269,000
Grader	kW	163	455,000	86,000	432,000

The cost of spare truck boxes and loader buckets is included in the capital cost for the major equipment cost estimate.

The distribution of capital costs is completed using the number of units required within a period. If new or replacement units are needed, that number of units and using the unit cost applied against them (20% of that for major equipment) is used to determine the capital cost in that period. There is no allowance for escalation in any of these costs.

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. Additional support equipment for snow removal and site water control was included to accommodate the expected climatic conditions.

The most significant piece of major mine equipment is the haulage trucks. At the peak of mining, seventeen 240 t units are necessary to maintain mine production. This happens from Year 5 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 situations, increasing the maximum on the number of trucks still leaves residual hours required to complete the material movement, therefore, the number of total trucks is unchanged. In these cases, the hours required are distributed evenly across the number of trucks on site and available. The other major mine equipment is determined in the same manner.

With a mine life of 11.3 years the major equipment does not require a replacement cycle. Support equipment is replaced within the mine life. 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 number of pieces of major equipment required by year are shown in Table 21-9.

Table 21-9: Mine Equipment on Site

Equipment	Yr -2	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12
Production Drill (140mm)	2	2	2	2	2	2	2	2	1	1	1	1	1	-
Production Drill (251mm)	-	-1	2	3	3	3	3	3	2	2	2	2	1	-
Production Loader	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Hydraulic Shovel (Electric)	-	2	2	2	2	2	2	2	2	2	2	2	2	-
Haulage Truck (91 mt)	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Haulage Truck (240 mt)	-	5	9	13	15	16	17	17	17	17	17	17	17	17
Crusher Loader	-	-	1	1	1	1	1	1	1	1	1	1	1	1
Production/Support Excavator	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Track Dozer	3	4	4	5	5	5	5	5	5	5	4	3	3	3
Grader	2	2	2	2	2	2	2	2	2	2	2	2	2	1

The smaller production drills, production excavator and haulage trucks (91 t) are used in the construction of the cofferdams, roads, and preparation of the WSF. After the main mine equipment arrives on site this equipment falls to a support role as required. The smaller drills are used for pre-shear drilling to keep the larger electric drills working on the more productive patterns. The production excavator will be used to clean the hanging wall and footwall of the ore zones to minimize dilution.

The expected equipment life is:

- production drill (140 mm) = 25,000 hrs

- production drill (251 mm) = 35,000 hrs
- hydraulic shovel (36 m^3) = 72,000 hrs
- production loader (23 m^3) = 50,000 hrs
- haulage truck (240 t) = 50,000 hrs
- haulage truck (91 t) = 35,000 hrs
- track dozer = 35,000 hrs
- grader = 25,000 hrs
- support excavator = 30,000 hrs

Other support equipment is normally determined in number of years and varies by its duty in the mine. Light plants for example are replaced every four years. The integrated tool carrier for site support is purchased once at the Project start and is not replaced over the mine life.

Ancillary Services (1700)

The ancillary services mine capital includes the mine offices and the fuel and lubrication depots. These have been separated from the other infrastructure capital as they relate to the mining activity. They are only initial costs, and no sustaining capital costs are considered. The mine offices are estimated at CDN$2.7 M and the fuel depot at CDN$1.0 M.

21.2.3 Site Development Capital

The site development capital comprises two main areas in the cost estimate:

- cofferdams (2100, 2200)
- bulk earthworks (2400)

The cofferdam cost totals CDN$12.5 M and includes the costs associated with the secant piling core and earthworks associated with that activity in the construction.

The bulk earthworks estimated in this area include:

- crusher earthworks
- plant site earthworks
- tailings loadout road and pad area
- internal roads
- surface water management infrastructure

The total cost of the bulk earthworks is CDN$15.5 M. No sustaining capital is included in this category.

21.2.4 Process Plant Capital

The process plant is designed for conventional process operations. The plant as estimated will treat 30,000 tpd or 1,250 tph. The process equipment requirements are based on the process flowsheet and process design criteria as defined in Section 17 of this Report.

Costs have been built up using the equipment lists for the Project, together with the layout and MTOs. The majority of costs for the equipment and construction are based on vendor budget quotations, with only 15% of the total being estimated or factored.

Labour costs are determined for each area using the local construction contractor's norm of seven days, 12 hours per day, 84 hours/working week, on a two week on, one week off rotational roster. A productivity factor has been included to account for site productivity inefficiencies such as meetings, labour skill, access difficulties, work complexity and site conditions. These factors are based on recent contractor submissions for similar projects in the region.

Concrete rates for supply and installation are based on budget pricing sourced from the market for supply and delivery of batched concrete including the supply and operation of the batch plant. Separate estimate packages were issued for the installation of concrete. Concrete costs in this estimate include material and installation and are based on all new concrete work.

The scope of the structural steel works allows for all new steel work in the process plant. All structural steel quantities were estimated from historical executed projects and factored accordingly to align with the process design. The steel quantities include light, medium, heavy, and extra heavy steel designations and miscellaneous steel including grating, handrail, and stair treads. Structural steel pricing is based on recently completed studies in Ontario and budget pricing was sourced from the market for supply and delivery to site of fabricated structural steel and other elements such as floor grating, handrailing, stair treads etc. A separate package was issued for installation of structural steel elements.

The scope of the architectural buildings work is noted below. Building sizes have been scaled-off general arrangement drawings and from similar historical projects. The architectural works covers the superstructure portion of the buildings. Overhead cranes, concrete works and internal support steel have been accounted for separately in their respective engineering discipline MTOs. The buildings cost estimate is based on the following list and building type:

1. Pre-engineered Metal-cladded Buildings
    a. primary crusher building
    b. grinding building
    c. process building
    d. gold room
    e. reagents building
2. Pre-engineered Fabric-cladded Buildings
    a. stockpile dome
3. Modular
    a. laboratory

Supply and install rates for the above superstructures are based on recently completed studies in Ontario. Budget pricing was sourced from the market for supply and installation packages for the architectural buildings.

Mechanical equipment estimates include the supply and installation of all new mechanical equipment identified in the PFS for the process plant. Equipment sizing has been determined by the process engineering and mechanical engineering teams, with a principle of selecting proven designs used to ensure equipment selection was robust. The mechanical equipment was specified using project-specific equipment datasheets highlighting agreed-upon process performance criteria and were accompanied by typical engineering specifications.

The items and quantities for the mechanical equipment list are based on the following:

- PFS process flow diagram
- equipment datasheets
- layouts and general arrangement drawings

The Mechanical Equipment List (MEL) has been populated, by the lead mechanical engineer, with ex-works supply pricing for every item. Pricing for the major mechanical equipment items have been based on budgetary quotations, using competitive pricing submissions from equipment suppliers. Where budgetary quotes were not obtained, existing database pricing has been used. For minor equipment, in-house historical pricing and estimates were used.

By value, 72% of mechanical equipment has been priced using budgetary supplier quotations, with 28% by estimates using historical data.

Piping cost estimates allow for the supply and installation of all pipework, fittings, valves, special pipe items and supports in the PFS for the process plant. Process plant piping has not been quantified. Process piping costs have been factored off the total installed process plant mechanical costs. The factors allow for pipe, fittings, supports, valves, paint, special pipe items and flanges. The overall factored piping equates to a blended 19.1% of total installed mechanical costs and aligns with historical benchmarks.

The estimate allows for the supply and installation of all the electrical equipment for the process plant and relative on-site facilities. An electrical equipment list (EEL) has been prepared and aligns with the current mechanical equipment list and load list. Electrical equipment pricing carried in the estimate has been based on budgetary vendor supply quotations for major equipment, and historical data from recent projects for minor equipment.

The electrical bulks estimate allows for the supply and installation of all electrical bulks in the PFS for the process plant. MTOs for electrical bulks have not been developed. Costs have been factored from the installed mechanical equipment costs for each process area. The factors are for cables, tray, terminations, wiring, testing, grounding, installation, etc. The overall electrical bulk costs equate to a blended 11.3% of total installed mechanical costs.

The instrumentation cost estimate allows for the supply and installation of all instruments and electrical bulks in the PFS for the process plant. An allowance for a plant control system has been included as well. MTOs for process instrumentation have not been developed. Costs have been factored from the installed mechanical equipment costs for each process area. The overall instrumentation factored costs equate to a blended 6.5% of total installed mechanical costs.

The process plant capital cost estimate, expressed in Canadian dollars, is shown in Table 21-10.

Table 21-10: Process Plant Capital Cost Estimate (CDN$)

WBS Code	Process Area	Total Cost, CDN$M
3100	Crushing, Stockpile, Reclaim	29.7
3200	Grinding	66.4
3300	Flotation	25.5
3400	Flotation, Tailings, Leaching	71.2
3500	Concentrate Leaching	60.3
3600	ADR Plant	15.0
3700	Refinery	7.2
3800	Reagents and Services	10.7
3900	Tailings Filtration	109.7
Total Process Plant Capital Costs		**395.7**

The process plant capital cost estimate is grouped into various categories of the WBS. Details of the plant design criteria are provided in Section 17 of this Report.

Crushing, Stockpile, Reclaim (3100)

The materials handling and crushing circuit cost includes the following key equipment:

- primary gyratory crusher
- mill feed apron feeders
- materials handling equipment

Grinding (3200)

The grinding circuit cost includes the following key equipment:

- SAG mill
- Ball mill
- cyclone feed pump box
- classification cyclone cluster
- trash screen

Flotation (3300)

The flotation circuit cost includes the following key equipment:

- rougher DFR feed head tank
- rougher DFRs
- rougher scavenger DFRs
- rougher flotation concentrate pump box
- cleaner scavenger DFR head tank

- cleaner scavenger DFRs
- cleaner flotation tails pump box
- cleaner flotation concentrate pump box
- flotation tails pump box

Flotation Tailings Leaching (3400)

The flotation tails leach and carbon adsorption circuit cost includes the following key equipment:

- trash screens
- leach/CIP tanks and agitators
- loaded carbon screens
- inter-tank carbon screens
- carbon sizing screens
- carbon safety screens

Concentrate Leaching (3500)

The flotation concentrate leach and carbon adsorption circuit cost includes the following key equipment:

- leach/CIP tanks and agitators
- loaded carbon screens
- inter-tank carbon screens
- carbon sizing screens
- carbon safety screens

ADR Plant (3600)

The main equipment in this cost includes:

- acid wash carbon columns
- stripping circuit cost includes the following key equipment:
  - elution columns
  - strip solution heater with heat exchanger
  - strip and pregnant solution tanks
  - carbon reactivation

Refinery (3700)

The electrowinning circuit and gold room costs include the following key equipment:

- electrowinning cells with rectifiers
- sludge pressure filter
- mercury retort
- flux mixer

- barring furnace with bullion moulds and slag handling system
- bullion vault and safe
- dust and fume collection and gas scrubbing system
- gold room security system

Reagents and Services (3800)

Cost for storage and preparation of key plant consumables include the following:

- pebble lime
- sodium cyanide
- sodium hydroxide
- hydrochloric acid
- copper sulphate (pentahydrate)
- sodium metabisulphite
- activated carbon
- flocculant
- coagulant
- flotation collector (PAX)
- flotation frother (MIBC)

Tailings Filtration (3900)

The main equipment in this cost includes:

- high-rate thickener
- overflow tank for process water storage
- underflow pumps
- filter feed tanks
- filter feed pumps
- pressure filters
- belt feeders
- loadout conveyor

21.2.5 On-site Infrastructure

Various items are included in the on-site infrastructure. This is all new capital. Estimates for the various areas under this category have been completed using the design criteria for the Project.

Power Supply and Distribution (4100)

The cost for the high voltage transmission line to site is included elsewhere. This cost category covers:

- main substation - purchase and installation (CDN$14.4 M)
- power reticulation (MV-LV)

- power generation and distribution
- camp transformer
- diesel gensets – standby (CDN$3.1 M)
- on-site powerlines (CDN$1.1 M)

The combined cost for these is estimated to be CDN$18.7 M.

Ancillary Buildings (4200)

Numerous buildings are required as part of the Project. These include:

- plant administration/office complex and dry (CDN$2.3 M)
- laboratory (CDN$2.2 M)
- plant workshop and warehouse (CDN$1.0 M)
- truck shop and wash bay (CDN$4.7 M)
- tire change shop (CDN$0.7 M)
- general buildings (CDN$2.2 M)
- gatehouse (CDN$0.1 M)

The total estimated cost of these buildings is CDN$13.3 M.

Site Services (4300)

Supporting a remote site such as the Project requires various services. These include:

- water storage and distribution (CDN$4.2 M)
- compressed air service (CDN$2.8 M)
- waste treatment and management (CDN$0.8 M)
- fuel storage and dispensing (CDN$0.1 M)
- fire detection and protection (CDN$2.2 M)
- water treatment plants (CDN$3.9 M)
- plant control systems (CDN$3.3 M)
- general site services (CDN$0.6 M)

The cost for this category (4300) has been estimated to be CDN$17.9 M

Mobile Equipment (4400)

Mobile equipment to support the site-wide and process operations has been included as part of the cost. This equipment includes:

- forklifts
- Hiab truck
- skid steer loader
- crew vans (15 person)
- first aid trailer

- scissor lifts
- crew cab pickup
- flat deck truck
- bucket truck

The cost for this equipment is expected to be CDN$1.3 M.

21.2.6 Off-Site Infrastructure Capital Cost

Costs within this category are those which are considered external to the main project area.

<u>Off-site Roads (5100)</u>

The main road to the site requires upgrading to allow heavy traffic to bring in the required materials for Project construction. This cost has been estimated to be CDN$0.5 M.

<u>Power Supply (OHPL) (5200)</u>

To meet the power requirements of the Project, a 230 kV line will have to be brought to the site. This will connect to the provincial grid's 230 kV line approximately 75 km to the southeast of the Project. The cost to bring this line in is estimated to be CDN$46.2 M.

<u>Water Supply (5300)</u>

In addition to the water reclaim from the filtered tails, and other ponds, make-up water will be required. This will come from Birch Lake. The cost for this water system is estimated at CDN$0.4 M.

21.2.7 Indirect Costs

Project indirect costs include all costs that are necessary for project completion but are not directly attributable to the construction of specific physical facilities of the plant or associated infrastructure but are required to be provided as support during the construction period. These items are as follows:

- field indirects
- camp accommodations and messing
- vendor representatives
- spares and first fills
- start-up and commissioning
- heavy equipment

<u>Field Indirects (6100)</u>

Field indirects are items or services provided by either the Owner or the EPCM contractor as common facilities and services which are not covered/managed by the general contractor's indirect costs (distributable).

These costs typically include but are not limited to:

- Temporary construction facilities – site office for EPCM contractor, site services, temporary fencing, temporary roads, and parking

- Construction support – site clean-up and waste disposal, material handling, maintenance of buildings and roads, testing and training, service labour, site transport, site surveys, QA/QC, and security
- Common pool of construction equipment, tools and supplies purchased by the Owner or the EPCM contractor – mobile equipment and tools, consumables, and purchased utilities
- Material transportation & storage on-site incurred by the Owner or the EPCM contractor – agents, staging and marshalling etc…

Field indirects have been factored from the total indirect costs (less mining) at a combined 3% and total CDN$15.7M.

Camp Accommodations and Messing (6200)

A construction camp has been allowed for in the estimate and will become the permanent operations accommodations upon completion of the construction. It is sized for a new 450-bed modular style construction camp including a commercial kitchen, dining/recreation room, and laundry facility. The camp cost is based on $40,000 per bed for construction.

In addition, a cost for operations and maintenance of the camp during the construction period of the Project has been included and is based on historical data at a cost of $50 per man per day.

Costs are summarized in Table 21-11.

Table 21-11: Construction Camp Costs

WBS	Area	Total Cost, CDN$M
6220	Construction Camp	18.0
6220	Camp Operations and Catering	8.4
Total Construction Camp Costs		**26.4**

Vendor Representatives (6400)

Costs for vendor representatives for construction and commissioning have been allowed for in the cost estimate and are based on historical percentages of 0.5% of the total supply cost of equipment, totalling CDN$1.4 M.

Spares and First Fills (6500)

Major mechanical spares for construction and commissioning have been included in the estimate, based on historical percentages of total equipment supply. Those costs are outlined in Table 21-12.

Table 21-12: Spares Cost Estimate

WBS	Description	Total Cost, CDN$M	Comments
6510	Commissioning Spares	1.4	1% of total equipment supply
6520	Capital (Critical) Spares	5.8	4% of total equipment supply
6530	Operating Spares	0.7	0.5% of total equipment supply
6540	Construction First Fills	0.7	0.5% of total equipment supply
6550	Commissioning First Fills	2.9	2% of total equipment supply
Total Spares and First Fills Costs		**11.5**	

<u>**Start-up and Commissioning (6600)**</u>

Commissioning assistance from mechanical completion to hand-over has been included in the estimate as an allowance of $2.6M.

A modification squad has been allowed for, the modification squad costs are monies allowed to provide construction contractors to assist the commissioning team to make minor modifications or provide labour assistance for commissioning. The modification squad value included in the estimate is calculated as 5 personnel for 60 days full time.

These costs are summarized in Table 12-13.

Table 21-13: Start-up and Commissioning Costs

WBS	Description	Total Cost, CDN$M	Comments
6610	Commissioning Spares	2.61	0.5% of direct costs less mining
6620	Capital (Critical) Spares	0.45	Based on 5 workers x 2 months fulltime
Total Start-up and Commissioning Costs		**3.06**	

<u>**Heavy Equipment (6700)**</u>

An allowance for heavy lift cranage (>100 t capacity cranage) has been included for extra-heavy lifts which are not included in the contractor's distributable costs (within the all-in SMP labour rates).

Heavy lift cranes, (i.e. generally cranes over 100t lifting capacity), and other high cost, low-utilization items of construction equipment, are often provided and managed by the implementation contractor via a common equipment pool to provide the opportunity to share and improve the utilization efficiency of these expensive items of equipment. This is estimated at 1% of total direct costs (less mining), plus an allowance for mobilization to a total of CDN$5.7M.

21.2.8 EPCM Cost

EPCM services costs cover such items as engineering and procurement services (home office based), construction management services (site based), project office facilities, information technology (IT), staff transfer expenses, secondary consultants, field inspection and expediting, corporate overhead and fees. An amount of $50M has been included in the cost estimate for EPCM services.

Mining EPCM services are included in their direct mining costs.

The breakout of the EPCM by WBS is shown in Table 12-14.

Table 21-14: EPCM Costs Summary

WBS	Area	Total Cost, CDN$M
7110	Engineering, Procurement	17.5
7120	Construction Management, Commissioning, 3rd Parties	27.5
7130	Expenses	5.0
In Mining Estimate	Mining	-
Total EPCM Costs		**50.0**

21.2.9 Owner's Cost Estimate

A placeholder in the cost estimate has been included on behalf of the client for PFS Level Owner's costs. An amount equivalent to 3% of total directs costs has been included and equates to CDN$21.4 M.

Owner's costs typically include, but are not limited to, the following:

- corporate overheads & office
- environmental monitoring
- site office
- setup & running costs
- staff & labour
- bonding

21.2.10 Contingency and Management Reserve

Contingency is a provision of funds for unforeseen or inestimable costs within the defined project scope relating to the level of engineering effort undertaken and estimate/engineering accuracy and applied to provide an overall level of confidence in costs and schedule outcomes (in this case, targeting a 50% confidence level). The contingency is meant to cover events or incidents that occur during the Project which cannot be quantified during the estimate preparation and does not include any allowance for project risk.

It is important to note that contingency does not cover scope changes, force majeure, adverse weather conditions, and changes in government policies, currency fluctuations, escalation, and other project risks.

A summary of the contingency contributors is noted in Table 21-15.

Table 21-15: Contingency Summary

Area	Comments	Total Cost, CDN$M
Process Plant, Infrastructure	15% of Plant and Infrastructure Costs	96.2
Mining Capital	5% of Mine Capital Costs	11.7
Owners Risk	Not Included	-
Total Contingency Costs		**107.9**

21.2.11 Closure Costs

Closure costs include covering of the waste rock storage area at the end of the mine life, noting that the current mine plan incorporates progressive reclamation during mining. Additional costs have been allocated for remediation of the depleted stockpiles, and the removal of the processing plant and site facilities in addition to the normal site closure costs.

Additional to the closure cost is the restoration of the lake area with the removal breaching of the cofferdams. Additional lake area is created within the mine quarry footprint upon closure and the lake level reaching its pre-mining level.

Mine closure costs have been estimated at CDN$39.3 M and will be complete in Year 13.

21.3 Operating Cost Estimates

21.3.1 Summary

The operating cost estimate is based on a combination of vendor quotations, first principal calculations, experience, reference projects, and factors as appropriate for a PFS. The operating costs by area are shown in Table 21-16 and Table 21-17.

Table 21-16: Operating Cost Estimate Summary (USD)

Operating Cost	Life of Mine Cost (USD$M)	Cost (USD$/t Processed)
Mining	793	6.52
Processing	1,323	10.87
G&A	96	0.79
TOTAL	**2,212**	**18.18**

Table 21-17: Operating Cost Estimate Summary (CDN$)

Operating Cost	Life of Mine Cost (CDN$M)	Cost (CDN$/t Processed)
Mining	1,057	8.69
Processing	1,763	14.50
G&A	129	1.06
TOTAL	**2,949**	**24.24**

21.3.2 Mining

The mine operating costs have been estimated from base principles with vendor quotations for repair and maintenance costs and other suppliers for consumables. Key inputs to the mine cost are fuel and labour. The price provided for the Project was CDN$0.80/L delivered to the site. The mine truck and support equipment fleets are diesel powered. The large production drills, hydraulic shovels and dewatering pumps are electric powered and used a price of CDN$0.08 per kilowatt hour.

Labour

Labour costs for the various job classifications were obtained from salary surveys in Ontario and other operations. A burden rate of 40% was applied to the various rates. Labour was estimated for both staff and hourly on a 12-hour shift basis utilizing a rotation of either two weeks on/two weeks off or 4 days on, three days off. Mine positions and salaries are shown in Table 21-18.

Table 21-18: Mine Staffing Requirements and Annual Employee Salaries (Year 5)

Position	Employees	Annual Salary (CDN$/a)
Mine Maintenance		
Maintenance Superintendent	1	210,000
Maintenance General Foreman	1	182,000
Maintenance Shift Foremen	4	147,000
Maintenance Planner/Contract Administration	2	133,000
Clerk	1	84,000
Subtotal	**9**	
Mine Operations		
Mine Operations/Technical Superintendent	1	224,000
Mine General Foreman	1	196,000
Senior Shift Foreman	4	147,000
Junior Shift Foreman	4	133,000
Road Crew/Services Foreman	1	147,000
Clerk	1	84,000
Subtotal	**12**	
Mine Engineering		
Chief Engineer	1	196,000
Senior Engineer	1	168,000
Open Pit Planning Engineer	2	147,000
Geotechnical Engineer	1	147,000
Blasting Engineer	1	147,000
Blasting/Geotechnical Technician	2	98,000
Dispatch Technician	1	98,000
Surveyor/Mining Technician	2	98,000
Surveyor/Mining Technician Helper	2	91,000
Clerk	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
Clerk	1	85,800
Subtotal	**9**	
TOTAL	**44**	

The mine staff labour remains constant from Year 2 until Year 8 when positions are removed as the mine winds down. During the pre-production period and Year 1 there are trainer positions in mine operations.

Hourly employee labour force levels in mine operations and maintenance fluctuate with production requirements. The Year 5 hourly labour requirements are shown in Table 21-19.

Table 21-19: Hourly Manpower Requirements and Annual Salaries (Year 5)

Position	Employees	Annual Salary ($/a)
Mine General		
General Equipment Operator	8	89,100
Road/Pump Crew	8	89,100
General Mine Labourer	8	88,600
Trainee	4	78,900
Light Duty Mechanic	3	150,900
Tire Technician	4	115,000
Lube Truck Driver	4	89,100
Subtotal	**39**	
Mine Operations		
Driller	16	119,000
Blaster	2	119,000
Blast Helper	4	88,600
Loader Operator	4	136,600
Hydraulic Shovel Operator	12	136,600
Haul Truck Driver	68	119,000
Dozer Operator	12	125,700
Grader Operator	6	125,700
Crusher Loader Operator	3	125,700
Snowplow/Water Truck	7	89,100
Subtotal	**134**	
Mine Maintenance		
Heavy Duty Mechanic	32	150,900
Welder	19	150,900
Electrician	2	150,900
Apprentice	7	108,900
Subtotal	**60**	
Total Hourly	**233**	

Labour costs are based on an Owner-operated scenario with First Mining responsible for the maintenance of the equipment with its own employees.

Overseeing all the mine operations, maintenance, engineering, and geology functions will be a Technical Superintendent. This person would have the Mine General Foreman and Maintenance Superintendent reporting to them, as well as the Chief Engineer and Chief Geologist.

The Mine General Foreman would have the shift foremen report directly to them.

The mine will have four mine operations crews, each with a Senior Shift Foremen who will have one Junior Shift Foreman reporting to them. For the mine life, there will also be 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 Training Foreman will only be required on site until the end of

Year 1, at which time the position will be eliminated. The Mine Operations department will have its own clerk/secretary.

The Chief Engineer will have one Senior Engineer and two open pit engineers reporting to them. The Blasting Engineer will be included in the short-range planning group and will double as Drill and Blast Foreman as required. The Geotechnical Engineer will cover all aspects of the wall slopes and waste dumps together with shared technicians in blasting.

The short-range planning group in engineering will also have 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 will be one Senior Geologist reporting to the Chief Geologist. There will also be 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 will also be four grade control/sampling technicians and one clerk/secretary.

Four Mine Maintenance Shift Foremen will report to the Maintenance General Foreman who in turn will report to the Maintenance Superintendent. As well, there will be two maintenance planners/contract administrators and a clerk.

The hourly labour force will include positions for the light duty mechanic, tire men, and lube truck drivers. These positions will all report to Maintenance. There will generally be one of each position per crew. Other general labour will include General Mine Labourers (two per crew) and Trainees (one per crew) plus two road/pump crew personnel per crew for water management/snow removal.

The drilling labour force is based on one operator per drill, per crew while operating. This peaks at 20 drillers in Year 6 and 7 and then drops down over time as the drilling hours are diminished.

Shovel and loader operators will peak at 16 in Year 1 and hold at that level until Year 8. Haulage truck drivers will peak at 68 in Year 5 and 6 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 that are required and are based on the number of equipment operators. Heavy duty mechanics work out to 0.25 mechanics required for each drill operator for example. Welders are 0.25 per operator and electricians are 0.05 per operator.

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.

Equipment Operating Costs

The vendors provided repair and maintenance (R&M) costs for each piece of equipment selected for the PFS. Fuel consumption rates were estimated from the supplied information and knowledge of the working conditions. The costs for the R&M are expressed in $/hr form.

Tire costs were also collected from various vendors for the sizes expected to be used. Estimates of tire life are based on AGP's experience and discussions with the vendors. The operating cost of the tires is also expressed in a $/hr form. The life of the haulage truck tires is estimated at 4,500 hours per tire with proper rotation from front to back. Each truck tire costs $35,000 so the cost per hour for tires is $46.67 /hr for the truck using six tires in the calculation.

Ground engaging tools (GET) costing is estimated from other projects and is an area that would be fine-tuned once the Project is operational.

Drill consumables are estimated as a complete drill string using the parts list and component lives provided by the vendor. Drill productivity is estimated at 25.7 m/hr for mill feed and waste. The equipment costs used in the estimate are shown in Table 21-20.

Table 21-20: Major Equipment Operating Costs – No labour (CDN$/hr)

Equipment	Fuel/ Power	Lube/Oil	Tires/ Undercarriage	Repair & Maintenance	GET/ Consumables	Total
Production Drill (140mm)	72.27	7.23	-	98.96	73.04	251.50
Production Drill (251mm)	20.00	-	6.00	75.80	69.65	171.45
Production Loader (23m^3)	88.33	8.83	67.20	121.13	15.00	300.49
Hydraulic Shovel (37m^3)	181.60	-	-	303.00	16.00	500.60
Haulage Truck (240 t)	117.24	11.72	46.67	107.00	6.00	288.63
Haulage Truck (91 t)	60.23	6.02	15.27	52.59	3.00	137.11
Track Dozer	60.23	6.02	10.00	64.04	5.00	145.29
Grader	17.67	1.77	4.00	18.58	5.00	47.01
Production Excavator	48.18	9.64	-	70.11	8.00	135.93

Drilling

Drilling in the open pit will use down the hole hammer drill rigs with 140 mm bits with the small diesel drill and rotary bits with the 251 mm electric drill. The material is designed to be blasted smaller and finer to improve productivity and reduce maintenance costs as well as improve plant performance. The drilling pattern parameters are shown in Table 21-21.

Table 21-21: Drill Pattern Specifications

Specification	Unit	Production Drill - Small			Production Drill – Large	
		Mill Feed	Waste	Pre-shear	Mill Feed	Waste
Bench Height	m	12	12	12	12	12
Sub-drill	m	0.8	0.8	0.0	1.30	1.30
Blasthole Diameter	mm	140	140	140	251	251
Pattern Spacing - Staggered	m	4.8	4.6	1.65	7.7	7.7
Pattern Burden – Staggered	m	4.2	4.0	2.0	6.7	6.7
Hole Depth	m	12.8	12.8	12.0	13.3	13.3

The sub-drill is included to allow for caving of the holes in weaker zones, reducing re-drill requirements or short holes that would affect bench floor conditions.

The parameters used to estimate drill productivity are shown in Table 21-22.

Table 21-22: Drill Productivity Criteria

Drill Activity	Unit	Small Drill	Production Drill
Pure Penetration Rate	m/min	0.55	0.50
Hole Depth	m	12.8	13.3
Drill Time	min	23.3	26.6
Move, Spot and Collar Hole	min	3.00	3.00
Level Drill	min	0.50	0.50
Add Steel	min	0.50	0.0
Pull Drill Rods	min	1.50	1.0
Total Setup/Breakdown Time	min	5.50	4.50
Total Drill Time per Hole	min	28.8	31.1
Drill Productivity	m/hr	26.7	25.7

Blasting

An emulsion product will be used for blasting to provide water protection. With the high rainfall, large snowmelt and working below lake level it is expected that a water-resistant explosive will be required. The powder factors used in the explosive calculation are shown in Table 21-23.

Table 21-23: Design Powder Factors

	Unit	Small Drill		Production Drill	
		Mill Feed	Waste	Mill Feed	Waste
Powder Factor	kg/m3	0.78	0.87	0.86	0.86
Powder Factor	kg/t	0.29	0.33	0.30	0.30

The blasting cost is estimated using quotations from a local explosives vendor. The emulsion price is $84.76/100 kg. The mine will be responsible for guiding the loading process, including placement of boosters/Nonels, and stemming and firing the shot.

The explosives vendor will also lease the explosives and accessories for a monthly cost. A service charge for the vendors pickup trucks, pumps, labour, and cost of the explosives plant are included. The total monthly cost is estimated at $136,000 per month.

Loading

Loading costs for both mill feed and waste are based on the use of electric hydraulic shovels and front-end loaders. The shovels will be the primary diggers with the front-end loaders as backup/support units. The average percentage of each material type that the various loading units are responsible for is shown in Table 21-24. This highlights the focus of the shovels over the loaders.

Table 21-24: Loading Parameters – Year 5

	Unit	Electric Hydraulic Shovel	Front End Loader
Bucket Capacity	m3	37	23
Truck Capacity Loaded	t	240	240
Waste Tonnage Loaded	%	85	15
Mill Feed Tonnage Loaded	%	75	25
Bucket Fill Factor	%	95	95
Cycle Time	sec	38	40
Trucks present at loading unit	%	80	80
Loading Time	min	2.60	4.70

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 loading unit. One of the largest operating cost items is haulage and minimizing this cost by maximizing the 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.

Hauling

Haulage profiles were determined for each pit phase for the primary crusher or the waste rock facility destinations. Cycle times were generated for the appropriate period tonnage by destination and phase to estimate the haulage costs. Maximum speed on the trucks is limited to 50 km/hr for tire life and safety reasons although few locations in the mine plan appeared to offer the truck the opportunity to accelerate to that velocity. Calculation speeds for various segments are shown in Table 21-25.

Table 21-25: Haulage Cycle Speeds

	Flat (0%) on surface	Flat (0%) In pit, Crusher, Dump	Slope Up (8%)	Slope Up (10%)	Slope Down (8%)	Slope Down (10%)
Loaded (km/hr.)	50	40	16	12.1	30	30
Empty (km/hr.)	50	40	35	25	35	35

Support Equipment

Support equipment hours and costs are determined on factors applied to various major pieces of equipment. For the PFS some of the factors used are shown in Table 21-26.

Table 21-26: Support Equipment Operating Factors

Mine Equipment	Factor	Factor Units
Track Dozer	25%	Of haulage hours to maximum of 5 dozers
Grader	15%	Of haulage hours to maximum of 2 graders
Crusher Loader	20%	Of loading hours to maximum of 1 loader
Snowplow/Water Truck	9%	Of haulage hours to maximum of 3 trucks
Pit Support Backhoe	8%	Of loading hours to maximum of 1 backhoe
Road Crew Backhoe	3	hours/day/unit
Road Crew Dump Truck	3	hours/day/unit
Road Crew Loader	3	hours/day/unit
Lube/Fuel Truck	6	hours/day/unit
Mechanics Truck	12	hours/day/unit
Blasting Loader	8	hours/day/unit
Blaster's Truck	8	hours/day/unit
Integrated Tool Carrier	4	hours/day/unit
Light Plants	12	hours/day/unit
Pickup Trucks	10	hours/day/unit

These factors resulted in the need for five track dozers, two graders, and one support backhoe. Their tasks include clean-up of the loader faces, roads, WSF, and blast patterns. The graders will maintain the crusher and waste haul routes. In addition, snowplows/water trucks have the responsibility for patrolling the haul roads for snow removal and controlling fugitive dust for safety and environmental reasons. The small backhoe and road crew dump trucks will be responsible for cleaning out sedimentation ponds and water ditch repairs.

The hours generated in this manner are applied to the individual operating costs for each piece of equipment. Many of these units are support equipment so no direct labour is allocated to them due to their variable function. The operators come from the General Equipment operator pool.

Grade Control

Grade control will be completed with a separate fleet of reverse circulation (RC) drill rigs. They will drill the deposit off on a 10 m x 5 m pattern in areas of known mineralization taking samples each metre. The holes will be inclined at 60°.

In areas of low-grade mineralization or waste the pattern spacing will be 20 m x 10 m with sampling over 6 m. These holes will be used to find undiscovered veinlets or pockets of mineralization. Over the life of the mine, a total of 917,000 m of drilling are expected to be completed for grade control work. A total of one million samples will be assayed from that drilling.

These grade control holes serve to define the mill feed grade and contacts.

Samples collected will be sent to the assay laboratory and assayed for use in the short range mining model.

No additional costing for blasthole sampling has been included. This may be an opportunity when more knowledge has been gained operationally on the gold deportment and representivity of blast hole samples.

Costs associated with this separate drill program are tracked as a distinct line item for the mining cost. The drill crew is one driller and two helpers with oversight by the Mine Geology department. The cost of this drilling is expected to average over $5 M per year.

Dewatering

Pit dewatering will be an important part of mining. Significant volumes need to be pumped initially to allow the open pit to advance as the bay is being dewatered, in addition to the normal rain/snow amounts.

Reviewing past data collected and comparing this to the proposed mining area allowed AGP to make an estimate at a PFS level for the water volume required to be pumped. Initial pumping in Year -1 is expected to be 1.8 Mm^3. That climbs to 6.2 Mm^3 per year in Year 9 as the pit drives deeper.

The dewatering will be completed with a set of six pumps in the pit and two pumps on the surface to push the water to the settling ponds. These pumps will be electric to reduce the cost of this operation.

Additional dewatering in the form of horizontal drill holes are part of the dewatering costs. These holes will be campaigned and will be part of the sustaining mine capital.

Dewatering is expected to cost $5.4 M over the mine life.

Leasing

Leasing of the mine fleet is considered a viable option to reduce initial capital. Various vendors offer this as an option to help select their equipment.

Indicative terms for leasing provided by the vendors are:

- down payment = 20% of equipment cost
- term length = 3-5 years (depending on equipment)
- 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 the term, equally distributed over those years. The calculation is as follows:

- annual lease cost = {[(initial capital cost) x 80%] x 1.067} / term in years

The initial capital, down payments, and annual leasing costs were shown previously in the capital cost area of this section.

The support equipment fleet is calculated in the same manner as the major mining equipment.

All of the major mine equipment, and the majority of the support equipment where it was considered reasonable, is leased. If the equipment has a life greater than the lease term length, then the following years onward of the lease do not have a lease payment applied. In the case of the mine trucks, with an approximate 10-year working life, the lease would be complete, 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 leases.

Utilizing the leasing option adds CDN$0.26/t moved to the mine operating cost over the life of the mine. On a cost per tonne of feed basis, it is CDN$1.18/t mill feed.

Total Mine Costs

The total life of mine operating costs per tonne of material moved (including rehandle and tailings backhaul) and per tonne of mill feed processed are shown in Table 21-27 and Table 21-28.

In the “General Mine Engineering” category is the cost associated with an Owner-operated crushing plant to make stemming material and road crush. That cost is approximately CDN$2.8 M per year.

Table 21-27: Open Pit Mine Operating Costs – with Leasing (CDN$/t Total Moved)

Open Pit Category	Unit	Year 1	Year 3	Year 5	LOM Average
General Mine and Engineering	$/t moved	0.19	0.17	0.17	0.18
Drilling	$/t moved	0.13	0.12	0.12	0.12
Blasting	$/t moved	0.30	0.33	0.34	0.30
Loading	$/t moved	0.16	0.16	0.15	0.16
Hauling	$/t moved	0.41	0.55	0.64	0.59
Support	$/t moved	0.19	0.20	0.20	0.22
Grade Control	$/t moved	0.08	0.12	0.08	0.10
Leasing Costs	$/t moved	0.42	0.45	0.24	0.26
Dewatering	$/t moved	0.00	0.01	0.01	0.01
Total	**$/t moved**	**1.88**	**2.11**	**1.96**	**1.94**

Table 21-28: Open Pit Mine Operating Costs – with Leasing (CDN$/t Mill Feed)

Open Pit Category	Unit	Year 1	Year 3	Year 5	LOM Average
General Mine and Engineering	$/t mill feed	1.08	0.95	0.95	0.80
Drilling	$/t mill feed	0.74	0.65	0.69	0.53
Blasting	$/t mill feed	1.70	1.84	1.90	1.36
Loading	$/t mill feed	0.92	0.86	0.86	0.70
Hauling	$/t mill feed	2.28	3.07	3.58	2.64
Support	$/t mill feed	1.05	1.11	1.13	0.97
Grade Control	$/t mill feed	0.45	0.69	0.47	0.46
Leasing Costs	$/t mill feed	2.33	2.52	1.34	1.18
Dewatering	$/t mill feed	0.03	0.06	0.07	0.05
Total	**$/t mill feed**	**10.58**	**11.75**	**10.97**	**8.69**

21.3.3 Processing

The annual process operating cost is an estimated CDN$158.8 M. A breakdown of this value and its unit costs is presented in Table 21-29.

Table 21-29: Average Annual Process Operating Cost

Cost Centre	Total Annual Costs $CDN M/a	CDN$/t	% of Total
Consumables	103	9.40	64.8%
Plant General Maintenance	7.69	0.70	4.8%
Power	30.21	2.76	19.0%
Laboratory	0.49	0.05	0.3%
Labour (O&M)	12.49	1.14	7.9%
Mobile Equipment	0.31	0.03	0.2%
Water Treatment	4.61	0.42	2.9%
Total	**158.8**	**14.50**	**100.0%**

Reagents and Operating Consumables

Individual reagent consumption rates were estimated based on metallurgical testwork results, experience, industry practice, and peer-reviewed literature. Reagent unit costs were obtained by benchmarking to similar projects.

Other consumables (e.g., primary crusher, SAG mill and ball mill liners, as well as grinding media for all mills) were estimated using:

- metallurgical testing results (i.e., abrasion tests)
- forecasted nominal power consumption
- supplier quotations

Reagents and consumables represent 64.8% of the total process operating cost at $9.40/t of plant feed.

Plant Maintenance

General plant maintenance costs were 5.0% of the total operating cost at CDN$7.69 M annually or CDN$0.70/t. Annual maintenance consumable costs were calculated based on a total installed mechanical cost by area using factors from 1.5 to 3%. The factor was applied to mechanical equipment, platework, and piping to generate the plant maintenance cost estimation.

Power

The processing power draw was based on the average power utilization of each motor on the electrical load list for the process plant and services. Power will be supplied by Ontario Hydro at CDN$0.08/kWh to service the facilities at the site. The total process plant power cost is CDN$30.21 M annually or CDN$2.76/t, 19.0% of the total process operating cost.

Mobile Equipment

Vehicle costs were based on a scheduled number of light vehicles and mobile equipment, including fuel, maintenance, spares and tires, and annual registration and insurance fees. The total process plant mobile equipment cost is CDN$0.31 M annually or CDN$0.03/t, 0.2% of the total process operating cost.

Labour

Staffing was estimated by benchmarking against similar projects. The labour costs incorporate requirements for plant operation, such as management, metallurgy, operations, maintenance, site services, assay laboratory, and contractor allowance. The total operational labour averages 99 employees.

Individual personnel were divided into their respective positions and classified as either 8-hour or 12-hour shift employees. Benefits were determined using published data from Mine Cost labour information for Ontario. The rates were estimated as overall rates, including all burden costs (benefits).

Water Treatment

Costs associated with treatment of water both from the plant and the settling ponds has been accumulated in this cost centre. It has been calculated to cost CDN$4.6 M per year or CDN$0.42/t.

21.3.4 General and Administrative Costs

A bottom-up approach was used to develop estimates for G&A costs over the LOM. The G&A costs were determined for a 12-year mine life with an average cost of CDN$1.06/t milled.

The G&A labour costs were estimated by developing a headcount profile for each department. Benefits were determined based on published data from Mine Costs' most recent database.

Health and safety equipment, camp and rotational travel, and human resource costs were determined by referencing recent projects. Environmental costs were determined based on the feedback from Auer. The IT and telecommunications costs for telecommunication, networking, internet, computers, radio system, and repairs were estimated as allowances.

A breakdown summary of forecast annual G&A costs is shown in Table 21-30.

Table 21-30: Annual Average G&A Operating Cost Summary

Cost Centre	Cost	
	CDN $/a	CDN $/t
Personnel	3,307,637	0.30
Camp	4,544,250	0.42
Rotational Travel Cost	858,600	0.08
Human Resources	181,920	0.02
Infrastructure Power	375,661	0.03
Site Admin, Maintenance and Security	67,860	0.01
Assets Operation	311,365	0.03
Health & Safety	225,960	0.02
Environmental	115,385	0.01
IT & Telecommunications	232,750	0.02
Contract Services	1,350,210	0.12
Total G&A Cost	**11,571,598**	**1.06**

22 ECONOMIC ANALYSIS

22.1 Cautionary Statement

The results of the economic analyses discussed in this section represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented herein. Information that is forward-looking includes the following:

- Mineral Resource and Mineral Reserve estimates
- assumed commodity prices and exchange rates
- proposed mine production plan
- projected mining and process recovery rates
- assumptions about mining dilution and the ability to mine in areas previously exploited using underground mining methods as envisaged
- sustaining costs and proposed operating costs
- interpretations and assumptions regarding joint venture and agreement terms
- assumptions as to closure costs and closure requirements
- assumptions about environmental, permitting, and social risks

Additional risks to the forward-looking information include:

- changes to costs of production from what is assumed
- unrecognized environmental risks
- unanticipated reclamation expenses
- unexpected variations in quantity of mineralized material, grade, or recovery rates
- geotechnical or hydrogeological considerations during mining being different from what was assumed
- failure of mining methods to operate as anticipated
- failure of plant, equipment, or processes to operate as anticipated
- changes to assumptions as to the availability of electrical power, and the power rates used in the operating cost estimates and financial analysis
- ability to maintain the social licence to operate
- accidents, labour disputes, and other risks of the mining industry
- changes to interest rates
- changes to tax rates

The mine plan is based on Probable Mineral Reserves that were converted from Indicated Mineral Resources.

Calendar years used in the financial analysis are provided for conceptual purposes only. Permits still have to be obtained in support of operations; and approval to proceed is still required from First Mining's Board of Directors.

22.2 Methodology Used

An economic model was developed to estimate annual pre-tax and post-tax cash flows and sensitivities of the Project based on a 5% discount rate. It must be noted that tax estimates involve complex variables that can only be accurately calculated during operations and, as such, the after-tax results are approximations. A sensitivity analysis was performed to assess the impact of variations in metal prices, head grades, initial capital cost, total operating cost, foreign exchange rate, and discount rate.

The capital and operating cost estimates developed specifically for this Project are presented in Section 21 of this report in Canadian dollars. The economic analysis has been run on a constant dollar basis with no inflation and the results converted to United States dollars for commonality in reporting against other projects.

22.3 Financial Model Parameters

Base case metal prices of USD$1,600/oz Au and USD$20/oz Ag are based on consensus analyst estimates and recently published economic studies. The forecasts are meant to reflect the average metal price expectation over the life of the Project. No price inflation or escalation factors were considered. Commodity prices can be volatile, and there is the potential for deviation from the forecast.

The economic analysis was performed using the following assumptions:

- construction starting January 1, 2024
- commercial production starting (effectively) on July 1, 2026, with mine production ramp-up, first revenue and expensed costs in Year +1
- mine life (LOM) of 11.3 years
- exchange rate of USD$0.75 per CDN$1.00
- cost estimates in constant Canadian dollars with no inflation or escalation
- 100% ownership with 1.3% NSR (assumes buy back of 1.4% NSR), see Section 22.6 for additional details
- capital costs funded with 100% equity (no financing costs assumed)
- all cash flows discounted to January 1, 2024 using end of year discounting convention
- gold is assumed to be sold in the same year it is produced
- no contractual arrangements for refining currently exist

22.4 Taxes

The Project has been evaluated on an after-tax basis to provide an approximate value of the potential economics. The tax model was compiled by First Mining with assistance from third-party taxation professionals. The calculations are based on the tax regime as of the date of the Pre-Feasibility Study and include estimates for First Mining's expenditures, and related impacts to various tax pool balances, between the Pre-Feasibility Study and the assumed construction start date.

At the effective date of this Report, the Project was assumed to be subject to the following tax regime:

- The Canadian corporate income tax system consists of 15% federal income tax and 10% provincial income tax.
- Ontario applies a mining tax rate of 10%.
- Total undiscounted tax payments are estimated to be USD$720 M over the life of mine.

22.5 Closure Costs

Total Project closure costs have been estimated to be USD$29 M, incurred partially in year 7 and with the remainder in the year after final production.

22.6 Royalty

Currently, the Project has an outstanding NSR royalty of 3%, but First Mining can buy back a significant portion of this amount prior to production. The financial model assumes the exercise of this option, which would result in the financial model carrying a 1.3% NSR. The financial model assumes USD$5.3 M consideration associated with the royalty buyback.

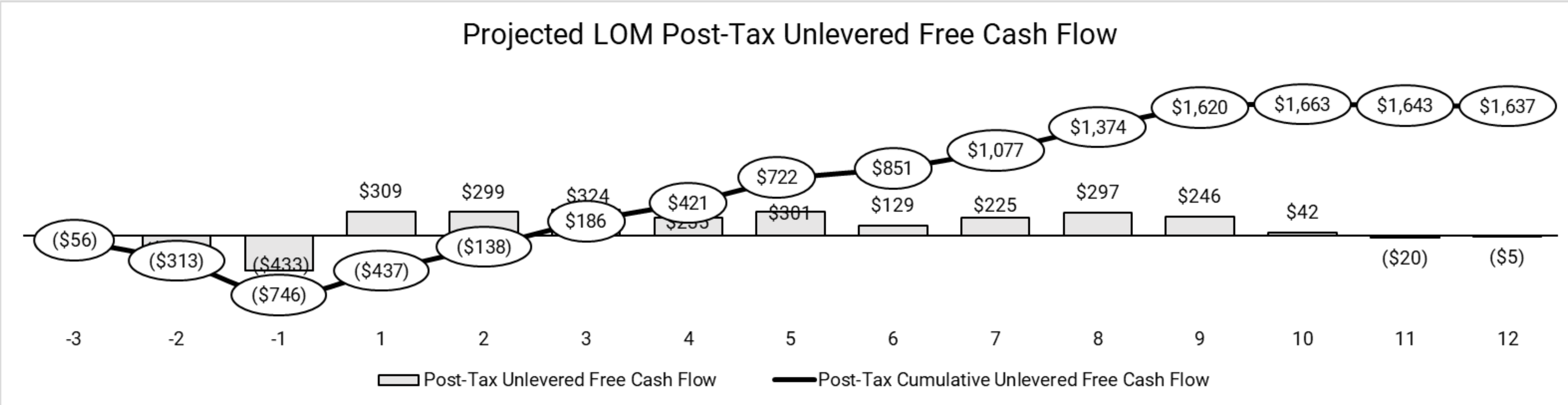
22.7 Silver Stream

Currently, the Project has a streaming agreement on 50% of the payable silver produced from Springpole over the life of the Project. First Mining can buy back half of this amount prior to production. The financial model assumes the exercise of this option for USD$22.5M prior to production, resulting in the model carrying 25% of the payable silver stream.

22.8 Economic Analysis

The economic analysis was performed assuming a 5% discount rate. The pre-tax NPV discounted at 5% is USD$1,482 M; the internal rate of return (IRR) is 36.4%; and payback period is 2.2 years. On an after-tax basis, the NPV discounted at 5% is USD$995 M; the IRR is 29.4%; and the payback period is 2.4 years. A summary of project economics is shown graphically in Figure 22-1 and listed in Table 22-1. Cashflow on an annualised basis is summarized in Table 22-2.

Figure 22-1: Projected Economics Graph



Source: AGP, 2020

Table 22-1: Summary of Project Economics

General	Units	LOM Total / Avg.
Gold Price	USD$/oz	$1,600
Silver Price	USD$/oz	$20.00
FX	CDN$:USD$	$0.75
Production		
Mine Life	yr.	11.3
Mined Ore	kt	121,636
Mined Waste	kt	287,532
Strip Ratio	w:o	2.36
Daily Throughput	tpd	30,000
Total Mill Feed	kt	121,636
Gold		
Mill Head Grade Au	g/t	0.97
Mill Recovery Au	%	85.7%
Total Payable Ounces Au	koz	3,225
Average Annual Payable Au	koz	287
Silver		
Mill Head Grade Ag	g/t	5.2
Mill Recovery Ag	%	89.5%
Total Payable Ounces Ag	koz	18,117
Average Annual Payable Ag	koz	1,610
Operating Cost		
Mining – mined	USD$/t mined	$2.06
Mining - milled	USD$/t milled	$6.52
Processing	USD$/t milled	$10.87
G&A	USD$/t milled	$0.79
Total	USD$/t milled	$18.18
Capital Cost		
Initial Capex	USD$M	$718
Sustaining Capex	USD$M	$55
Closure Cost	USD$M	$29
Operating Costs per Ounce		
Cash Costs (net)	USD$/oz	$618
AISC (net)	USD$/oz	$645
Cash Costs	USD$/oz AuEq	$673
AISC	USD$/oz AuEq	$698
Pre-Tax Economics		
NPV (5%)	USD$M	$1,482
IRR	%	36.4%
Post-Tax Economics		
NPV (5%)	USD$M	$995
IRR	%	29.4%
Payback	yr.	2.4

Notes: * Cash costs consist of mining costs, processing costs, mine-level G&A and refining charges and royalties. ** AISC includes cash costs plus sustaining capital and closure costs. AISC is at a project-level and does not include an estimate of corporate G&A.

Table 22-2: Project Cash Flow on an Annualised Basis

Dollar figures in Real 2020 USD$M unless otherwise noted	Inputs / Sensitivity	Units	Total / Avg.	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13
Cash Flow Summary																			
Cash flows discounted to January 01, 2024																			
Attributable Gross Revenue		**USD$M**	**$5,522**	**-**	**-**	**-**	**$531**	**$603**	**$676**	**$559**	**$661**	**$405**	**$546**	**$631**	**$538**	**$208**	**$135**	**$29**	**-**
Silver Stream		USD$M	($61)	-	-	-	($3)	($6)	($6)	($6)	($7)	($4)	($6)	($9)	($7)	($4)	($3)	($1)	-
Royalties Paid		USD$M	($74)	-	-	-	($10)	($9)	($9)	($7)	($7)	($5)	($7)	($9)	($7)	($2)	($2)	($0)	-
Transportation & Refining Charges		USD$M	($10)	-	-	-	($1)	($1)	($1)	($1)	($1)	($1)	($1)	($1)	($1)	($0)	($0)	($0)	-
Operating Costs		USD$M	($2,212)	-	-	-	($191)	($221)	($224)	($221)	($218)	($216)	($213)	($195)	($172)	($157)	($150)	($33)	-
EBITDA		**USD$M**	**$3,165**	**-**	**-**	**-**	**$327**	**$366**	**$435**	**$324**	**$428**	**$179**	**$319**	**$417**	**$351**	**$44**	**($20)**	**($5)**	**-**
Initial Capex		USD$M	($718)	($56)	($257)	($405)	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capex		USD$M	($55)	-	-	-	($18)	($10)	($2)	($8)	($8)	($7)	($0)	($1)	($1)	-	-	-	-
Closure		USD$M	($29)	-	-	-	-	-	-	-	-	-	($3)	-	-	-	-	-	($27)
Royalty Buyback		USD$M	($5)	-	-	($5)	-	-	-	-	-	-	-	-	-	-	-	-	-
Stream Buyback		USD$M	($23)	-	-	($23)	-	-	-	-	-	-	-	-	-	-	-	-	-
Pre-Tax Unlevered Free Cash Flow	**$2,323**	**USD$M**	**($0.0)**	**($56)**	**($257)**	**($433)**	**$309**	**$357**	**$433**	**$316**	**$420**	**$172**	**$315**	**$416**	**$350**	**$44**	**($20)**	**($5)**	**($27)**
Pre-Tax Cumulative Unlevered Free Cash Flow		USD$M	$2,334	*($56)*	*($313)*	*($746)*	*($437)*	*($81)*	*$352*	*$667*	*$1,087*	*$1,259*	*$1,575*	*$1,991*	*$2,341*	*$2,386*	*$2,366*	*$2,361*	*$2,324*
Cash Taxes (Income +Mining Taxes)		USD$M	**($723)**	**-**	**-**	**-**	**-**	**($57)**	**($109)**	**($81)**	**($119)**	**($43)**	**($90)**	**($119)**	**($103)**	**($2)**	**-**	**-**	**-**
Post-Tax Unlevered Free Cash Flow	**$1,596**	**USD$M**	**($0.0)**	**($56)**	**($257)**	**($433)**	**$309**	**$299**	**$324**	**$235**	**$301**	**$129**	**$225**	**$297**	**$246**	**$42**	**($20)**	**($5)**	**($27)**
Post-Tax Cumulative Unlevered Free Cash Flow		USD$M	*$1,611*	*($56)*	*($313)*	*($746)*	*($437)*	*($138)*	*$186*	*$421*	*$722*	*$851*	*$1,077*	*$1,374*	*$1,620*	*$1,663*	*$1,643*	*$1,637*	*$1,611*
Pricing Assumptions																			
Gold Price	$1,600	USD$/oz	$1,600	$1,600	$1,600	$1,600	$1,600	$1,600	$1,600	$1,600	$1,600	$1,600	$1,600	$1,600	$1,600	$1,600	$1,600	$1,600	$1,600
Silver Price	$20.00	USD$/oz	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00
Exchange Rate	$0.75	CDN$:USD$	$0.75	$0.75	$0.75	$0.75	$0.75	$0.75	$0.75	$0.75	$0.75	$0.75	$0.75	$0.75	$0.75	$0.75	$0.75	$0.75	$0.75
Production																			
Production Summary																			
Total Material Mined		**kt**	**409,168**	**-**	**3,862**	**20,318**	**39,567**	**50,000**	**50,000**	**50,000**	**50,137**	**50,000**	**50,000**	**26,617**	**9,971**	**6,779**	**1,918**	**-**	**-**
Mill Feed																			
Total Mill Feed		**kt**	59,274	-	-	-	8,669	5,868	4,696	3,665	959	4,149	6,341	6,580	6,696	4,880	5,584	1,186	-
Payable Gold																			
Total Au Head Grade		g/t	0.97	-	-	-	1.17	1.16	1.30	1.06	1.27	0.78	1.07	1.22	1.04	0.48	0.30	0.30	-
Total Au Recovery		%	85.7%	-	-	-	88.5%	87.8%	88.3%	88.3%	87.3%	87.4%	86.0%	85.0%	85.0%	70.0%	70.0%	70.0%	-
Total Au Payability		%	99.0%	-	-	-	99.0%	99.0%	99.0%	99.0%	99.0%	99.0%	99.0%	99.0%	99.0%	99.0%	99.0%	99.0%	-
Total Payable Gold		**koz**	**3,225**	**-**	**-**	**-**	**322**	**354**	**400**	**328**	**387**	**237**	**320**	**361**	**310**	**116**	**74**	**16**	**-**
Payable Silver																			
Total Ag Head Grade		g/t	5.23	-	-	-	3.20	5.74	5.64	5.55	6.62	4.32	5.42	8.37	6.54	3.69	2.76	2.76	-
Total Ag Recovery		%	89.5%	-	-	-	83.0%	90.3%	90.5%	90.3%	91.8%	87.6%	89.8%	92.0%	91.8%	85.0%	81.4%	81.4%	-
Total Ag Payability		%	99.0%	-	-	-	99.0%	99.0%	99.0%	99.0%	99.0%	99.0%	99.0%	99.0%	99.0%	99.0%	99.0%	99.0%	-
Total Payable Gold - 100%		**koz**	**18,117**	**-**	**-**	**-**	**824**	**1,805**	**1,779**	**1,746**	**2,125**	**1,318**	**1,698**	**2,683**	**2,099**	**1,092**	**782**	**166**	**-**
Revenue																			
Total Payable AuEq		**koz AuEq**	**3,413**	**-**	**-**	**-**	**330**	**373**	**419**	**346**	**409**	**251**	**338**	**389**	**332**	**128**	**82**	**18**	**-**
Total Au Revenue		**USD$M**	**$5,159**	**-**	**-**	**-**	**$515**	**$567**	**$640**	**$524**	**$619**	**$379**	**$512**	**$577**	**$496**	**$186**	**$119**	**$25**	**-**
Total Ag Revenue – Spot		**USD$M**	**$272**	**-**	**-**	**-**	**$12**	**$27**	**$27**	**$26**	**$32**	**$20**	**$25**	**$40**	**$31**	**$16**	**$12**	**$2**	**-**
Total Ag Revenue - Streamed		**USD$M**	**$30**	**-**	**-**	**-**	**$1**	**$3**	**$3**	**$3**	**$4**	**$2**	**$3**	**$4**	**$3**	**$2**	**$1**	**$0**	**-**

Total Attributable Gross Revenue		**USD$M**	**$5,461**	**-**	**-**	**-**	**$529**	**$597**	**$670**	**$553**	**$654**	**$401**	**$540**	**$622**	**$531**	**$204**	**$132**	**$28**	**-**
Net Smelter Return																			
Pre-Stream Revenue		USD$M	$5,522	-	-	-	$531	$603	$676	$559	$661	$405	$546	$631	$538	$208	$135	$29	-
Less: Transportation & Refining Charges	$3.00/oz AuEq	USD$M	$10	-	-	-	$1	$1	$1	$1	$1	$1	$1	$1	$1	$0	$0	$0	-
Net Smelter Return		**USD$M**	**$5,511**	**-**	**-**	**-**	**$530**	**$602**	**$674**	**$558**	**$660**	**$405**	**$545**	**$630**	**$537**	**$208**	**$134**	**$29**	**-**
Operating Costs																			
Mining		CDN$M	$1,057	-	-	-	$103	$124	$129	$124	$120	$118	$114	$89	$59	$40	$29	$8	-
Processing		CDN$M	$1,763	-	-	-	$141	$159	$159	$159	$159	$159	$159	$159	$159	$159	$159	$34	-
G&A		CDN$M	$129	-	-	-	$10	$12	$12	$12	$12	$12	$12	$12	$12	$12	$12	$2	-
Total Operating Costs		**CDN$M**	**$2,949**	**-**	**-**	**-**	**$255**	**$294**	**$299**	**$294**	**$291**	**$288**	**$285**	**$259**	**$230**	**$210**	**$200**	**$44**	**-**
Base Case Costs																			
Mining		*CDN$/t moved*	*$1.94*	-	-	-	*$1.88*	*$2.03*	*$2.11*	*$1.96*	*$1.96*	*$1.85*	*$1.87*	*$2.37*	*$2.46*	*$1.40*	*$1.23*	*$1.67*	-
Processing Inc. Water Treatment		*CDN$/t milled*	*$14.50*	-	-	-	*$14.50*	*$14.50*	*$14.50*	*$14.50*	*$14.50*	*$14.50*	*$14.50*	*$14.50*	*$14.50*	*$14.50*	*$14.50*	*$14.50*	-
G&A		*CDN$/t milled*	*$1.06*	-	-	-	*$1.06*	*$1.06*	*$1.06*	*$1.06*	*$1.06*	*$1.06*	*$1.06*	*$1.06*	*$1.06*	*$1.06*	*$1.06*	*$1.06*	-
Operating Costs per Tonne Processed		*CDN$/t milled*	$24.24	-	-	-	*$26.13*	*$26.87*	*$27.30*	*$26.89*	*$26.53*	*$26.32*	*$25.99*	*$23.69*	*$20.91*	*$19.17*	*$18.22*	*$18.90*	-
Operating Costs - USD$																			
Mining		*USD$M*	$793	-	-	-	$77	$93	$96	$93	$90	$88	$86	$67	$44	$30	$22	$6	-
Processing Inc. Water Treatment		*USD$M*	$1,323	-	-	-	$106	$119	$119	$119	$119	$119	$119	$119	$119	$119	$119	$25	-
G&A		*USD$M*	$96	-	-	-	$8	$9	$9	$9	$9	$9	$9	$9	$9	$9	$9	$2	-
Total Operating Costs		**USD$M**	**$2,212**	**-**	**-**	**-**	**$191**	**$221**	**$224**	**$221**	**$218**	**$216**	**$213**	**$195**	**$172**	**$157**	**$150**	**$33**	**-**
Mining		*USD$/t moved*	$1.46	-	-	-	$1.41	$1.52	$1.58	$1.47	$1.47	$1.38	$1.41	$1.78	$1.84	$1.05	$0.92	$1.26	-
Mining		*USD$/t mined*	$2.06	-	-	-	$1.95	$1.86	$1.93	$1.86	$1.80	$1.77	$1.71	$2.51	$4.42	$4.38	$11.42	-	-
Processing		*USD$/t milled*	$10.87	-	-	-	$10.87	$10.87	$10.87	$10.87	$10.87	$10.87	$10.87	$10.87	$10.87	$10.87	$10.87	$10.87	-
G&A		*USD$/t milled*	$0.79	-	-	-	$0.79	$0.79	$0.79	$0.79	$0.79	$0.79	$0.79	$0.79	$0.79	$0.79	$0.79	$0.79	-
Operating Costs per Tonne Processed		*USD$/t milled*	$18.18	--	--	-	$19.60	$20.15	$20.48	$20.17	$19.90	$19.74	$19.49	$17.77	$15.68	$14.38	$13.67	$14.18	-
Cash Costs / AISC																			
Co-Product																			
Cash Cost (Co-Product Basis) *		*USD$/oz AuEq*	$673	-	-	-	$612	$618	$561	$662	$554	$884	$656	$527	$543	$1,252	$1,839	$1,907	-
All-in Sustaining Cost (Co-Product Basis)**		*USD$/oz AuEq*	$698	-	-	-	$666	$644	$567	$687	$573	$913	$666	$529	$546	$1,252	$1,839	$1,907	-
By-Product																			
Cash Cost (By-Product Basis) *		*USD$/oz Au*	$618	-	-	-	$585	$566	$513	$610	$494	$843	$604	$444	$469	$1,218	$1,865	$1,940	-
All-in Sustaining Cost (By-Product Basis)**		*USD$/oz Au*	$645	-	-	-	$641	$594	$519	$636	$514	$873	$614	$446	$471	$1,218	$1,865	$1,940	-

* Cash costs consist of mining costs, processing costs, mine-level G&A and refining charges and royalties

** AISC includes cash costs plus sustaining capital and closure costs

Capital Expenditures																			

Capital Expenditures - CDN$																		
1000 Mining	CDN$M	$193	$10	$74	$108	-	-	-	-	-	-	-	-	-	-	-	-	-
2000 Site Development	CDN$M	$28	$5	$23	-	-	-	-	-	-	-	-	-	-	-	-	-	-
3000 Process Plant	CDN$M	$396	-	$99	$297	-	-	-	-	-	-	-	-	-	-	-	-	-
4000 On-Site Infrastructure	CDN$M	$51	$4	$36	$11	-	-	-	-	-	-	-	-	-	-	-	-	-
5000 Off-Site Infrastructure	CDN$M	$47	$12	$18	$17	-	-	-	-	-	-	-	-	-	-	-	-	-
6000 Indirects	CDN$M	$64	$8	$20	$35	-	-	-	-	-	-	-	-	-	-	-	-	-
7000 EPCM Services	CDN$M	$50	$10	$20	$20	-	-	-	-	-	-	-	-	-	-	-	-	-
8000 Owner's Costs	CDN$M	$21	$4	$9	$9	-	-	-	-	-	-	-	-	-	-	-	-	-
9000 Provisions	CDN$M	$108	$22	$43	$43	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Initial Capital	**CDN$M**	**$958**	**$75**	**$342**	**$540**													
1000 Mining	CDN$M	$68	-	-	-	$24	$13	$3	$11	$5	$10	$0	$1	$1	-	-	-	-
2000 Site Development	CDN$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
3000 Process Plant	CDN$M	$6	-	-	-	-	-	-	-	$6	-	-	-	-	-	-	-	-
4000 On-Site Infrastructure	CDN$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
5000 Off-Site Infrastructure	CDN$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
6000 Indirects	CDN$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
7000 EPCM Services	CDN$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
8000 Owner's Costs	CDN$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
9000 Provisions	CDN$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Sustaining Capital (Excludes Closure)	**CDN$M**	**$74**	**-**	**-**	**-**	**$24**	**$13**	**$3**	**$11**	**$10**	**$10**	**$0**	**$1**	**$1**	**-**	**-**	**-**	**-**
Total Closure Cost	**CDN$M**	**$39**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**$4**	**-**	**-**	**-**	**-**	**-**	**$35**
Capital Expenditures - USD$																		
1000 Mining	USD$M	$144	$8	$56	$81	-	-	-	-	-	-	-	-	-	-	-	-	-
2000 Site Development	USD$M	$21	$4	$17	-	-	-	-	-	-	-	-	-	-	-	-	-	-
3000 Process Plant	USD$M	$297	-	$74	$223	-	-	-	-	-	-	-	-	-	-	-	-	-
4000 On-Site Infrastructure	USD$M	$38	$3	$27	$8	-	-	-	-	-	-	-	-	-	-	-	-	-
5000 Off-Site Infrastructure	USD$M	$35	$9	$13	$13	-	-	-	-	-	-	-	-	-	-	-	-	-
6000 Indirects	USD$M	$48	$6	$15	$27	-	-	-	-	-	-	-	-	-	-	-	-	-
7000 EPCM Services	USD$M	$38	$8	$15	$15	-	-	-	-	-	-	-	-	-	-	-	-	-
8000 Owner's Costs	USD$M	$16	$3	$6	$6	-	-	-	-	-	-	-	-	-	-	-	-	-
9000 Provisions	USD$M	$81	$16	$32	$32	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Initial Capital	**USD$M**	**$718**	**$56**	**$257**	**$405**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
1000 Mining	USD$M	$51	-	-	-	$18	$10	$2	$8	$4	$7	$0	$1	$1	-	-	-	-
2000 Site Development	USD$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
3000 Process Plant	USD$M	$4	-	-	-	-	-	-	-	$4	-	-	-	-	-	-	-	-
4000 On-Site Infrastructure	USD$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
5000 Off-Site Infrastructure	USD$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
6000 Indirects	USD$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
7000 EPCM Services	USD$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
8000 Owner's Costs	USD$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
9000 Provisions	USD$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Sustaining Capital (Excludes Closure)	**USD$M**	**$55**	**-**	**-**	**-**	**$18**	**$10**	**$2**	**$8**	**$8**	**$7**	**$0**	**$1**	**$1**	**-**	**-**	**-**	**-**
Total Closure Cost	**USD$M**	**$29**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**$3**	**-**	**-**	**-**	**-**	**-**	**$27**

22.9 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and after-tax NPV and IRR of the Project, using the following variables: metal prices, head grades, initial capital cost, total operating cost, foreign exchange rate, and discount rate. Table 22-3 provides a summary of the sensitivity analysis.

Table 22-4 shows the Project's pre-tax sensitivity, and Table 22-5 shows the Project's post-tax sensitivity results.

Figure 22-2 and Figure 22-3 show the sensitivity in a spider graph format.

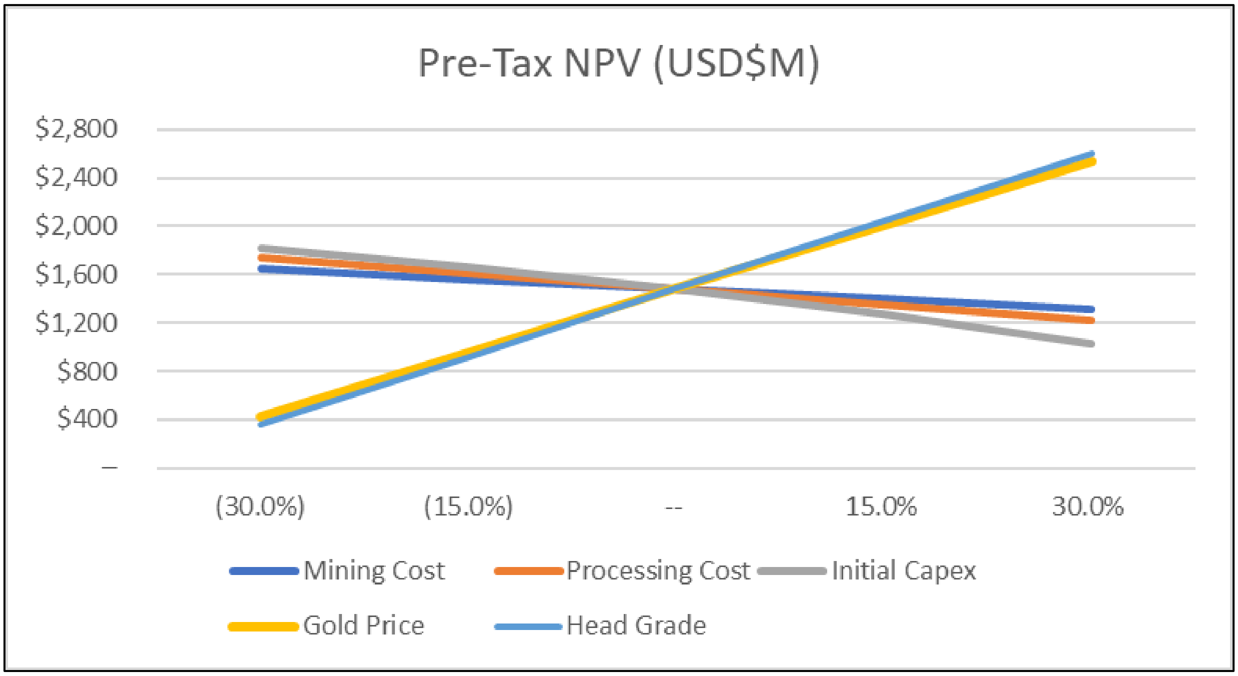
The analysis revealed that the Project is most sensitive to revenue attributes such as gold price, head grade and exchange rate, followed by operating cost and capital cost.

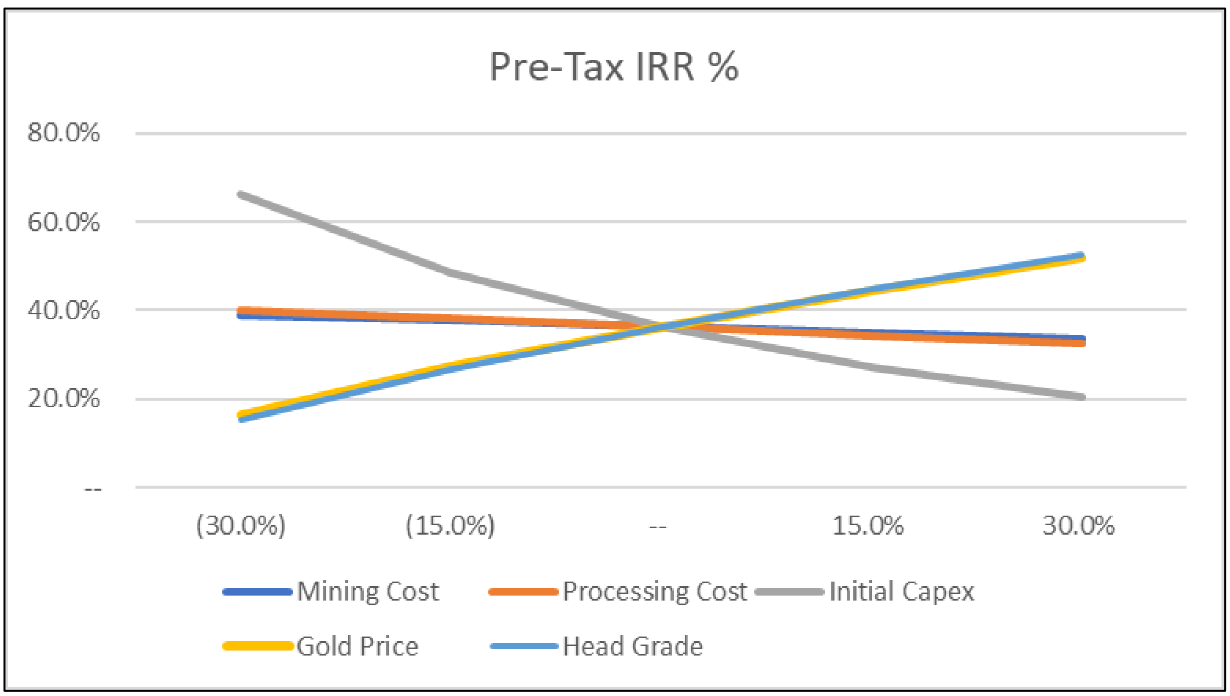
Table 22-3: Sensitivity Summary

Post-Tax NPV (USD$M)						
	Base	(30.0%)	(15.0%)	--	15.0%	30.0%
Mining Cost	$995	$1,109	$1,052	$995	$938	$880
Processing Cost	$995	$1,175	$1,086	$995	$902	$809
Initial Capex	$995	$1,277	$1,151	$995	$810	$596
Au Price	$995	$260	$629	$995	$1,357	$1,718
Head Grade	$995	$216	$607	$995	$1,378	$1,759
Post-Tax IRR						
	Base	(30.0%)	(15.0%)	--	15.0%	30.0%
Mining Cost	29.4%	31.6%	30.6%	29.4%	28.3%	27.2%
Processing Cost	29.4%	32.4%	31.0%	29.4%	27.9%	26.2%
Initial Capex	29.4%	55.9%	40.4%	29.4%	21.4%	15.1%
Au Price	29.4%	13.1%	22.0%	29.4%	36.1%	42.0%
Head Grade	29.4%	11.9%	21.6%	29.4%	36.4%	42.5%

Pre-Tax NPV (USD$M)						
	Base	(30.0%)	(15.0%)	--	15.0%	30.0%
Mining Cost	$1,482	$1,647	$1,564	$1,482	$1,399	$1,316
Processing Cost	$1,482	$1,745	$1,613	$1,482	$1,350	$1,218
Initial Capex	$1,482	$1,814	$1,663	$1,482	$1,271	$1,031
Au Price	$1,482	$426	$954	$1,482	$2,009	$2,537
Head Grade	$1,482	$363	$922	$1,482	$2,040	$2,598
Pre-Tax IRR						
	Base	(30.0%)	(15.0%)	--	15.0%	30.0%
Mining Cost	36.4%	39.1%	37.7%	36.4%	35.0%	33.6%
Processing Cost	36.4%	40.0%	38.2%	36.4%	34.5%	32.5%
Initial Capex	36.4%	66.1%	48.6%	36.4%	27.3%	20.3%
Au Price	36.4%	16.6%	27.3%	36.4%	44.5%	51.9%
Head Grade	36.4%	15.2%	26.8%	36.4%	44.8%	52.5%

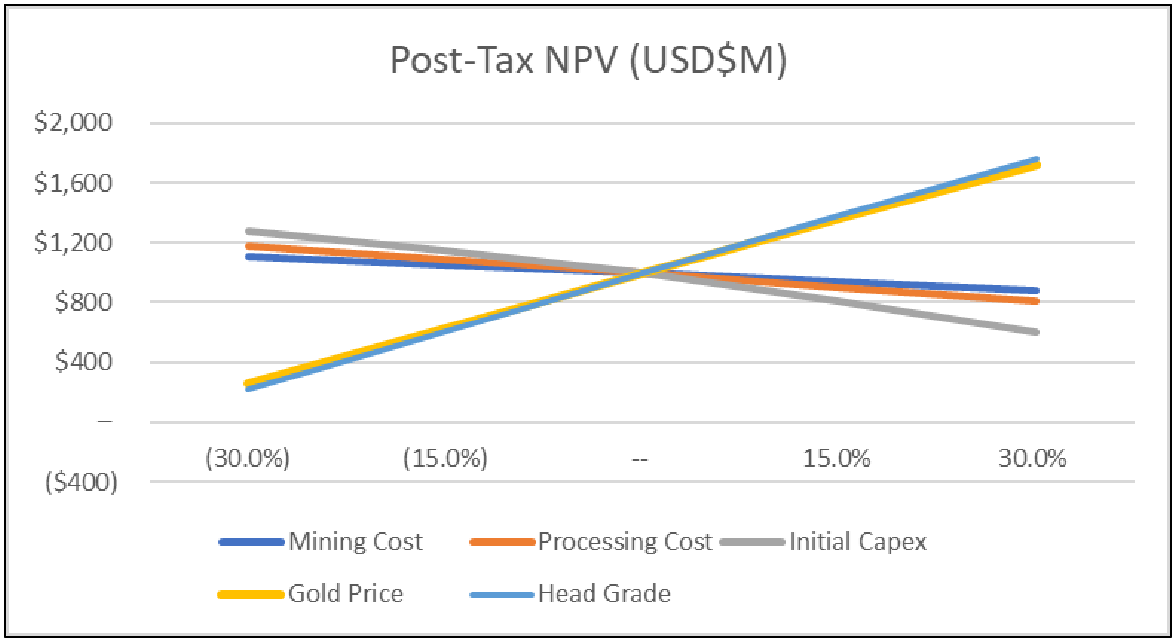
Figure 22-2 Pre-Tax Spider Graphs

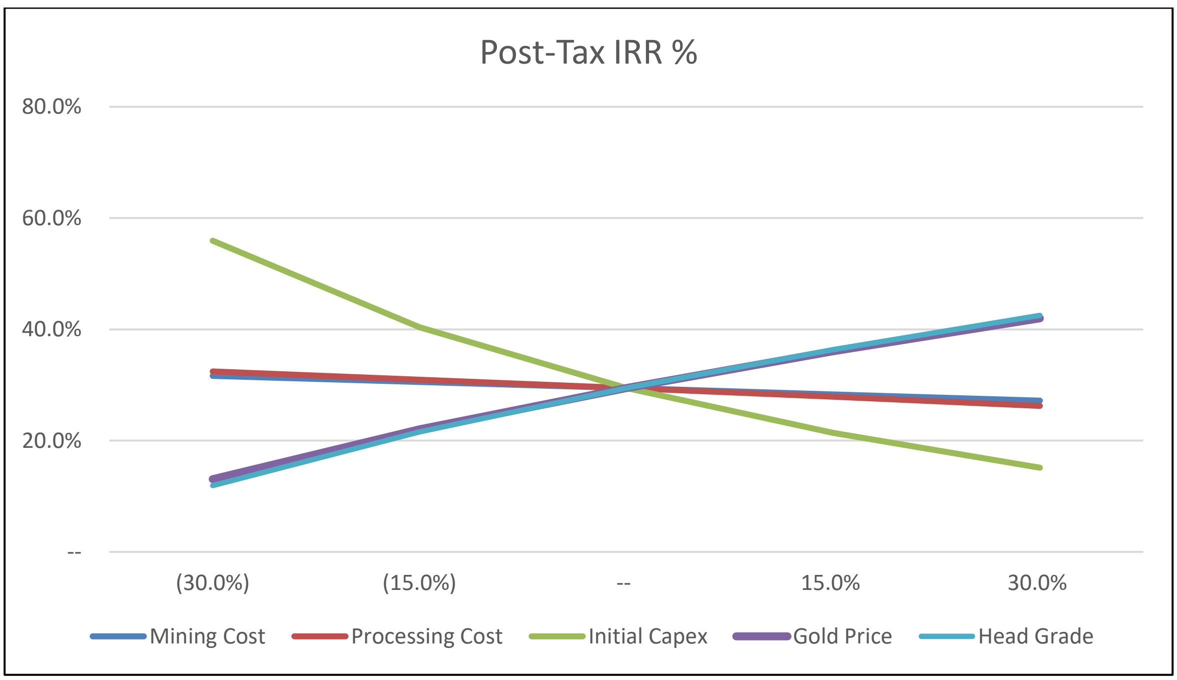




Source: AGP, 2021

Figure 22-3: Post-Tax Spider Graphs





Source: AGP, 2021

Table 22-4: Pre-Tax Sensitivity

Pre-Tax NPV (USD$M) Sensitivity To Discount Rate

	Au Price (USD$/oz)								
	$1,482	$1,300	$1,400	$1,500	**$1,600**	$1,700	$1,800	$1,900	$2,000
Discount Rate	1.0%	$1,246	$1,541	$1,835	$2,130	$2,424	$2,719	$3,014	$3,308
	3.0%	$1,014	$1,268	$1,522	$1,775	$2,029	$2,283	$2,537	$2,790
	5.0%	$822	$1,042	$1,262	**$1,482**	$1,701	$1,921	$2,141	$2,361
	8.0%	$592	$771	$951	$1,130	$1,309	$1,488	$1,667	$1,846
	10.0%	$470	$627	$785	$942	$1,099	$1,257	$1,414	$1,571

Pre-Tax IRR % Sensitivity To Discount Rate

	Au Price (USD$/oz)								
	36.4%	$1,300	$1,400	$1,500	**$1,600**	$1,700	$1,800	$1,900	$2,000
Discount Rate	1.0%	24.8%	28.9%	32.7%	36.4%	39.8%	43.2%	46.4%	49.5%
	3.0%	24.8%	28.9%	32.7%	36.4%	39.8%	43.2%	46.4%	49.5%
	5.0%	24.8%	28.9%	32.7%	**36.4%**	39.8%	43.2%	46.4%	49.5%
	8.0%	24.8%	28.9%	32.7%	36.4%	39.8%	43.2%	46.4%	49.5%
	10.0%	24.8%	28.9%	32.7%	36.4%	39.8%	43.2%	46.4%	49.5%

Pre-Tax NPV (USD$M) Sensitivity To Mining Cost

	Au Price (USD$/oz)								
	$1,482	$1,300	$1,400	$1,500	**$1,600**	$1,700	$1,800	$1,900	$2,000
Mining Cost	(20%)	$932	$1,152	$1,372	$1,592	$1,812	$2,032	$2,251	$2,471
	(10%)	$877	$1,097	$1,317	$1,537	$1,757	$1,976	$2,196	$2,416
	--	$822	$1,042	$1,262	**$1,482**	$1,701	$1,921	$2,141	$2,361
	10%	$767	$987	$1,206	$1,426	$1,646	$1,866	$2,086	$2,306
	20%	$712	$931	$1,151	$1,371	$1,591	$1,811	$2,031	$2,251

Pre-Tax IRR Sensitivity To Mining Cost

	Au Price (USD$/oz)								
	36.4%	$1,300	$1,400	$1,500	**$1,600**	$1,700	$1,800	$1,900	$2,000
Mining Cost	(20%)	27.0%	30.9%	34.6%	38.2%	41.6%	44.8%	48.0%	51.0%
	(10%)	25.9%	29.9%	33.7%	37.3%	40.7%	44.0%	47.2%	50.2%
	--	24.8%	28.9%	32.7%	**36.4%**	39.8%	43.2%	46.4%	49.5%
	10%	23.7%	27.9%	31.8%	35.5%	39.0%	42.3%	45.6%	48.7%
	20%	22.6%	26.9%	30.8%	34.5%	38.1%	41.5%	44.7%	47.9%

Pre-Tax NPV (USD$M) Sensitivity To Processing Cost

	Au Price (USD$/oz)								
	$1,482	$1,300	$1,400	$1,500	**$1,600**	$1,700	$1,800	$1,900	$2,000
Processing Cost	(20%)	$997	$1,217	$1,437	$1,657	$1,877	$2,097	$2,317	$2,536
	(10%)	$910	$1,130	$1,349	$1,569	$1,789	$2,009	$2,229	$2,449
	--	$822	$1,042	$1,262	**$1,482**	$1,701	$1,921	$2,141	$2,361
	10%	$734	$954	$1,174	$1,394	$1,614	$1,834	$2,053	$2,273
	20%	$646	$866	$1,086	$1,306	$1,526	$1,746	$1,966	$2,186

Pre-Tax IRR Sensitivity To Processing Cost

	Au Price (USD$/oz)								
		$1,300	$1,400	$1,500	**$1,600**	$1,700	$1,800	$1,900	$2,000
Processing Cost	(20%)	27.8%	31.6%	35.3%	38.8%	42.2%	45.4%	48.5%	51.5%
	(10%)	26.3%	30.3%	34.0%	37.6%	41.0%	44.3%	47.4%	50.5%
	--	24.8%	28.9%	32.7%	**36.4%**	39.8%	43.2%	46.4%	49.5%
	10%	23.3%	27.5%	31.4%	35.1%	38.7%	42.0%	45.3%	48.4%
	20%	21.6%	26.0%	30.0%	33.8%	37.4%	40.9%	44.2%	47.3%

Pre-Tax NPV (USD$M) Sensitivity To Initial Capex

	Au Price (USD$/oz)								
	$1,482	$1,300	$1,400	$1,500	**$1,600**	$1,700	$1,800	$1,900	$2,000
Initial Capex	(20%)	$1,057	$1,277	$1,497	$1,716	$1,936	$2,156	$2,376	$2,596
	(10%)	$946	$1,166	$1,386	$1,605	$1,825	$2,045	$2,265	$2,485
	--	$822	$1,042	$1,262	**$1,482**	$1,701	$1,921	$2,141	$2,361
	10%	$685	$905	$1,125	$1,345	$1,564	$1,784	$2,004	$2,224
	20%	$535	$755	$975	$1,194	$1,414	$1,634	$1,854	$2,074

Pre-Tax IRR Sensitivity To Initial Capex

	Au Price (USD$/oz)								
		$1,300	$1,400	$1,500	**$1,600**	$1,700	$1,800	$1,900	$2,000
Initial Capex	(20%)	39.2%	44.3%	49.2%	53.7%	58.1%	62.4%	66.4%	70.4%
	(10%)	31.2%	35.8%	40.0%	44.1%	48.0%	51.7%	55.3%	58.7%
	--	24.8%	28.9%	32.7%	**36.4%**	39.8%	43.2%	46.4%	49.5%
	10%	19.5%	23.3%	26.8%	30.1%	33.2%	36.2%	39.1%	41.9%
	20%	15.1%	18.6%	21.8%	24.8%	27.7%	30.4%	33.1%	35.6%

Pre-Tax NPV Sensitivity To Head Grade % Change

	Au Price (USD$/oz)								
	$1,482	$1,300	$1,400	$1,500	**$1,600**	$1,700	$1,800	$1,900	$2,000
Head Grade % Change	(20%)	$208	$384	$560	$736	$912	$1,088	$1,263	$1,439
	(10%)	$515	$713	$911	$1,109	$1,307	$1,505	$1,702	$1,900
	--	$822	$1,042	$1,262	**$1,482**	$1,701	$1,921	$2,141	$2,361
	10%	$1,128	$1,370	$1,612	$1,854	$2,096	$2,338	$2,579	$2,821
	20%	$1,435	$1,698	$1,962	$2,226	$2,490	$2,754	$3,018	$3,281

Pre-Tax IRR Sensitivity To Head Grade % Change

	Au Price (USD$/oz)								
		$1,300	$1,400	$1,500	**$1,600**	$1,700	$1,800	$1,900	$2,000
Head Grade % Change	(20%)	11.2%	15.6%	19.6%	23.2%	26.6%	29.8%	32.9%	35.8%
	(10%)	18.6%	22.7%	26.6%	30.2%	33.6%	36.8%	39.9%	42.9%
	--	24.8%	28.9%	32.7%	**36.4%**	39.8%	43.2%	46.4%	49.5%
	10%	30.4%	34.5%	38.4%	42.1%	45.7%	49.1%	52.4%	55.6%
	20%	35.5%	39.7%	43.7%	47.5%	51.1%	54.6%	58.0%	61.3%

Pre-Tax NPV (USD$M) Sensitivity To Metal Price

	Au Price (USD$/oz)								
	$1,482	$1,300	$1,400	$1,500	**$1,600**	$1,700	$1,800	$1,900	$2,000
Silver Price	$16.25	$785	$1,005	$1,225	$1,445	$1,664	$1,884	$2,104	$2,324
	$17.50	$797	$1,017	$1,237	$1,457	$1,677	$1,897	$2,116	$2,336
	$18.75	$810	$1,029	$1,249	$1,469	$1,689	$1,909	$2,129	$2,349
	$20.00	$822	$1,042	$1,262	**$1,482**	$1,701	$1,921	$2,141	$2,361
	$21.25	$834	$1,054	$1,274	$1,494	$1,714	$1,934	$2,153	$2,373
	$22.50	$847	$1,066	$1,286	$1,506	$1,726	$1,946	$2,166	$2,386
	$23.75	$859	$1,079	$1,298	$1,518	$1,738	$1,958	$2,178	$2,398
	$25.00	$870	$1,090	$1,310	$1,530	$1,750	$1,970	$2,190	$2,409

Pre-Tax IRR Sensitivity To Metal Price

	Au Price (USD$/oz)								
		$1,300	$1,400	$1,500	**$1,600**	$1,700	$1,800	$1,900	$2,000
Silver Price	$16.25	24.2%	28.3%	32.2%	35.9%	39.4%	42.7%	45.9%	49.0%
	$17.50	24.4%	28.5%	32.4%	36.0%	39.5%	42.9%	46.1%	49.2%
	$18.75	24.6%	28.7%	32.6%	36.2%	39.7%	43.0%	46.2%	49.3%
	$20.00	24.8%	28.9%	32.7%	**36.4%**	39.8%	43.2%	46.4%	49.5%
	$21.25	25.0%	29.1%	32.9%	36.5%	40.0%	43.3%	46.5%	49.6%
	$22.50	25.2%	29.3%	33.1%	36.7%	40.2%	43.5%	46.6%	49.7%
	$23.75	25.4%	29.5%	33.3%	36.9%	40.3%	43.6%	46.8%	49.9%
	$25.00	25.6%	29.7%	33.4%	37.0%	40.5%	43.7%	46.9%	50.0%

Table 22-5: Post-Tax Sensitivity

Post-Tax NPV (USD$M) Sensitivity To Discount Rate

	Au Price (USD$/oz)								
	$995	$1,300	$1,400	$1,500	**$1,600**	$1,700	$1,800	$1,900	$2,000
Discount Rate	1.0%	$849	$1,055	$1,260	$1,463	$1,666	$1,868	$2,071	$2,273
	3.0%	$678	$856	$1,032	$1,207	$1,382	$1,556	$1,730	$1,904
	5.0%	$537	$690	$843	**$995**	$1,146	$1,297	$1,448	$1,599
	8.0%	$367	$492	$617	$740	$864	$987	$1,110	$1,232
	10.0%	$276	$387	$496	$604	$713	$821	$929	$1,037

Post-Tax IRR % Sensitivity To Discount Rate

	Au Price (USD$/oz)								
	29.4%	$1,300	$1,400	$1,500	**$1,600**	$1,700	$1,800	$1,900	$2,000
Discount Rate	1.0%	20.0%	23.3%	26.5%	29.4%	32.3%	35.0%	37.6%	40.1%
	3.0%	20.0%	23.3%	26.5%	29.4%	32.3%	35.0%	37.6%	40.1%
	5.0%	20.0%	23.3%	26.5%	**29.4%**	32.3%	35.0%	37.6%	40.1%
	8.0%	20.0%	23.3%	26.5%	29.4%	32.3%	35.0%	37.6%	40.1%
	10.0%	20.0%	23.3%	26.5%	29.4%	32.3%	35.0%	37.6%	40.1%

Post-Tax NPV (USD$M) Sensitivity To Mining Cost

	Au Price (USD$/oz)								
	$995	$1,300	$1,400	$1,500	**$1,600**	$1,700	$1,800	$1,900	$2,000
Mining Cost	(20%)	$614	$767	$919	$1,071	$1,222	$1,373	$1,524	$1,674
	(10%)	$575	$729	$881	$1,033	$1,184	$1,335	$1,486	$1,636
	--	$537	$690	$843	**$995**	$1,146	$1,297	$1,448	$1,599
	10%	$498	$651	$805	$957	$1,108	$1,259	$1,410	$1,561
	20%	$460	$613	$767	$919	$1,070	$1,221	$1,372	$1,523

Post-Tax IRR Sensitivity To Mining Cost

	Au Price (USD$/oz)								
	29.4%	$1,300	$1,400	$1,500	**$1,600**	$1,700	$1,800	$1,900	$2,000
Mining Cost	(20%)	21.7%	25.0%	28.0%	30.9%	33.7%	36.3%	38.9%	41.3%
	(10%)	20.8%	24.2%	27.3%	30.2%	33.0%	35.7%	38.2%	40.7%
	--	20.0%	23.3%	26.5%	**29.4%**	32.3%	35.0%	37.6%	40.1%
	10%	19.0%	22.5%	25.7%	28.7%	31.6%	34.3%	36.9%	39.5%
	20%	18.1%	21.6%	24.9%	28.0%	30.9%	33.6%	36.3%	38.8%

Post-Tax NPV (USD$M) Sensitivity To Processing Cost

	Au Price (USD$/oz)								
	$995	$1,300	$1,400	$1,500	**$1,600**	$1,700	$1,800	$1,900	$2,000
Processing Cost	(20%)	$661	$815	$965	$1,116	$1,267	$1,416	$1,566	$1,716
	(10%)	$599	$753	$905	$1,055	$1,206	$1,357	$1,508	$1,657
	--	$537	$690	$843	**$995**	$1,146	$1,297	$1,448	$1,599
	10%	$474	$627	$781	$933	$1,085	$1,237	$1,388	$1,539
	20%	$412	$565	$719	$871	$1,024	$1,176	$1,327	$1,478

Post-Tax IRR Sensitivity To Processing Cost

	Au Price (USD$/oz)								
	29.4%	$1,300	$1,400	$1,500	**$1,600**	$1,700	$1,800	$1,900	$2,000
Processing Cost	(20%)	22.4%	25.6%	28.6%	31.5%	34.2%	36.8%	39.3%	41.7%
	(10%)	21.2%	24.5%	27.6%	30.5%	33.3%	35.9%	38.5%	40.9%
	--	20.0%	23.3%	26.5%	**29.4%**	32.3%	35.0%	37.6%	40.1%
	10%	18.6%	22.1%	25.4%	28.4%	31.3%	34.1%	36.7%	39.2%
	20%	17.2%	20.9%	24.2%	27.3%	30.3%	33.1%	35.8%	38.4%

Post-Tax NPV (USD$M) Sensitivity To Initial Capex

	Au Price (USD$/oz)								
	$995	$1,300	$1,400	$1,500	**$1,600**	$1,700	$1,800	$1,900	$2,000
Initial Capex	(20%)	$739	$891	$1,044	$1,196	$1,347	$1,497	$1,648	$1,798
	(10%)	$644	$798	$950	$1,102	$1,253	$1,404	$1,555	$1,705
	--	$537	$690	$843	**$995**	$1,146	$1,297	$1,448	$1,599
	10%	$416	$569	$723	$875	$1,026	$1,177	$1,328	$1,479
	20%	$283	$436	$589	$743	$894	$1,044	$1,195	$1,346

Post-Tax IRR Sensitivity To Initial Capex

	Au Price (USD$/oz)								
	29.4%	$1,300	$1,400	$1,500	**$1,600**	$1,700	$1,800	$1,900	$2,000
Initial Capex	(20%)	33.0%	37.2%	41.1%	44.9%	48.5%	51.8%	55.1%	58.2%
	(10%)	25.8%	29.5%	33.0%	36.3%	39.5%	42.5%	45.4%	48.1%
	--	20.0%	23.3%	26.5%	**29.4%**	32.3%	35.0%	37.6%	40.1%
	10%	15.1%	18.2%	21.1%	23.8%	26.4%	28.9%	31.2%	33.5%
	20%	11.1%	14.0%	16.6%	19.1%	21.5%	23.7%	25.9%	28.0%

Post-Tax NPV Sensitivity To Head Grade % Change

	Au Price (USD$/oz)								
	$995	$1,300	$1,400	$1,500	**$1,600**	$1,700	$1,800	$1,900	$2,000
Head Grade % Change	(20%)	$106	$231	$355	$477	$599	$723	$845	$966
	(10%)	$323	$461	$599	$737	$874	$1,011	$1,147	$1,283
	--	$537	$690	$843	**$995**	$1,146	$1,297	$1,448	$1,599
	10%	$751	$918	$1,084	$1,251	$1,416	$1,582	$1,747	$1,911
	20%	$962	$1,143	$1,325	$1,506	$1,686	$1,865	$2,044	$2,224

Post-Tax IRR Sensitivity To Head Grade % Change

	Au Price (USD$/oz)								
	29.4%	$1,300	$1,400	$1,500	**$1,600**	$1,700	$1,800	$1,900	$2,000
Head Grade % Change	(20%)	8.6%	12.3%	15.6%	18.6%	21.4%	24.1%	26.6%	29.0%
	(10%)	14.8%	18.2%	21.4%	24.4%	27.1%	29.8%	32.4%	34.8%
	--	20.0%	23.3%	26.5%	**29.4%**	32.3%	35.0%	37.6%	40.1%
	10%	24.6%	27.9%	31.1%	34.2%	37.0%	39.8%	42.4%	44.9%
	20%	28.7%	32.2%	35.4%	38.5%	41.4%	44.2%	46.9%	49.5%

Post-Tax NPV (USD$M) Sensitivity To Metal Price

	Au Price (USD$/oz)								
	$995	$1,300	$1,400	$1,500	**$1,600**	$1,700	$1,800	$1,900	$2,000
Silver Price	$16.25	$511	$665	$818	$970	$1,121	$1,272	$1,423	$1,574
	$17.50	$520	$673	$826	$978	$1,129	$1,280	$1,431	$1,582
	$18.75	$528	$682	$835	$986	$1,138	$1,289	$1,440	$1,590
	$20.00	$537	$690	$843	**$995**	$1,146	$1,297	$1,448	$1,599
	$21.25	$545	$699	$851	$1,003	$1,154	$1,305	$1,456	$1,607
	$22.50	$554	$707	$860	$1,012	$1,163	$1,314	$1,465	$1,615
	$23.75	$562	$716	$868	$1,020	$1,171	$1,322	$1,473	$1,623
	$25.00	$570	$724	$876	$1,028	$1,179	$1,330	$1,481	$1,631

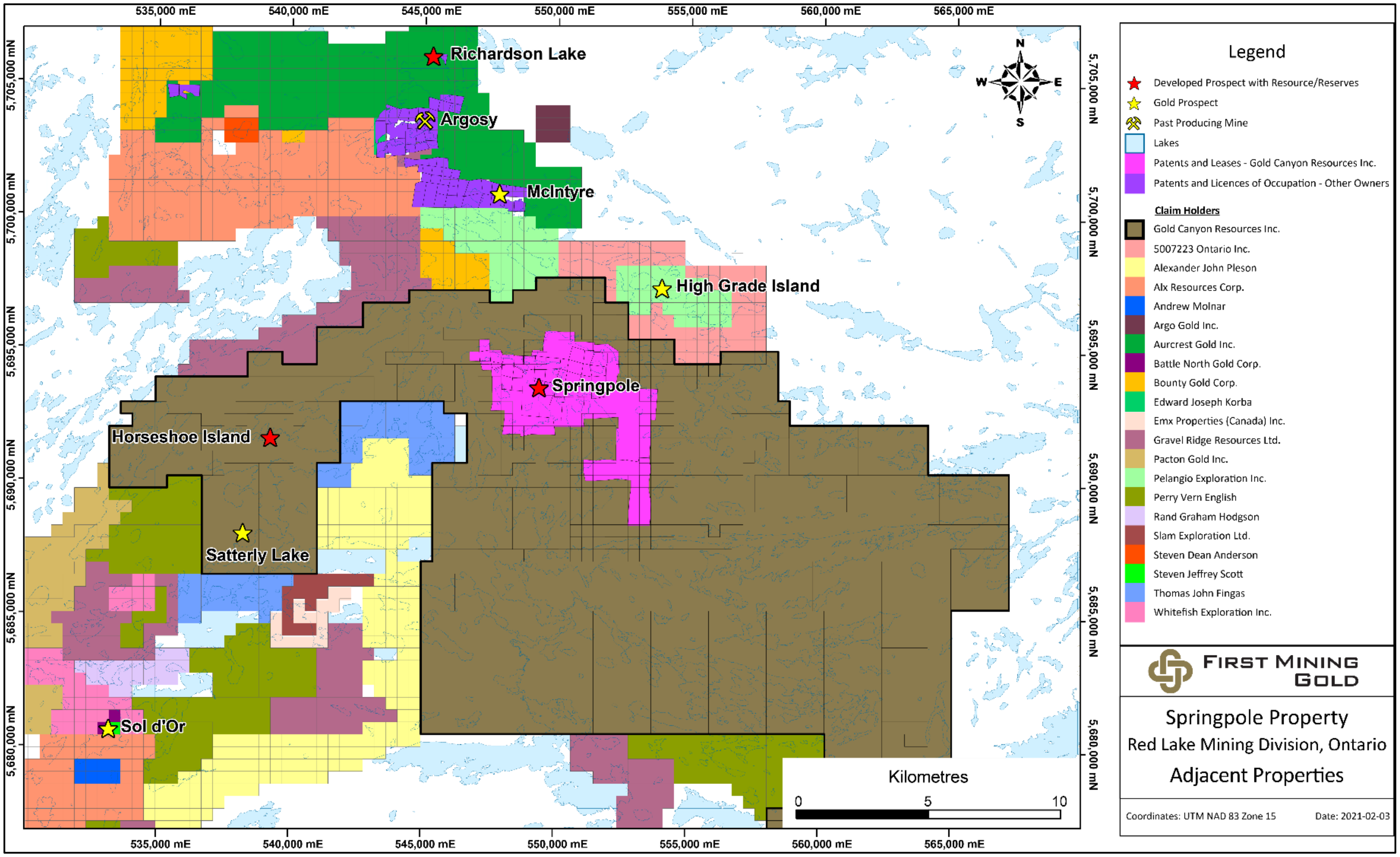
Post-Tax IRR Sensitivity To Metal Price

	Au Price (USD$/oz)								
	29.4%	$1,300	$1,400	$1,500	**$1,600**	$1,700	$1,800	$1,900	$2,000
Silver Price	$16.25	19.4%	22.9%	26.0%	29.0%	31.9%	34.6%	37.2%	39.7%
	$17.50	19.6%	23.0%	26.2%	29.2%	32.0%	34.8%	37.4%	39.9%
	$18.75	19.8%	23.2%	26.3%	29.3%	32.2%	34.9%	37.5%	40.0%
	$20.00	20.0%	23.3%	26.5%	**29.4%**	32.3%	35.0%	37.6%	40.1%
	$21.25	20.1%	23.5%	26.6%	29.6%	32.4%	35.1%	37.7%	40.2%
	$22.50	20.3%	23.7%	26.8%	29.7%	32.6%	35.3%	37.8%	40.3%
	$23.75	20.5%	23.8%	26.9%	29.8%	32.7%	35.4%	37.9%	40.4%
	$25.00	20.6%	24.0%	27.0%	30.0%	32.8%	35.5%	38.0%	40.5%

23 ADJACENT PROPERTIES

There are several mineral exploration properties adjacent to the Springpole property covering portions of the Birch-Uchi greenstone belt. The area has seen sporadic exploration over the past 90 years and hosts several past-producing mines (Figure 23-1).

Figure 23-1: Adjacent Properties Map



Source: First Mining, 2021
(Note: Tenure information sourced from the Ontario Mining Lands Administration System (MLAS), February 2021)

24 OTHER RELEVANT DATA AND INFORMATION

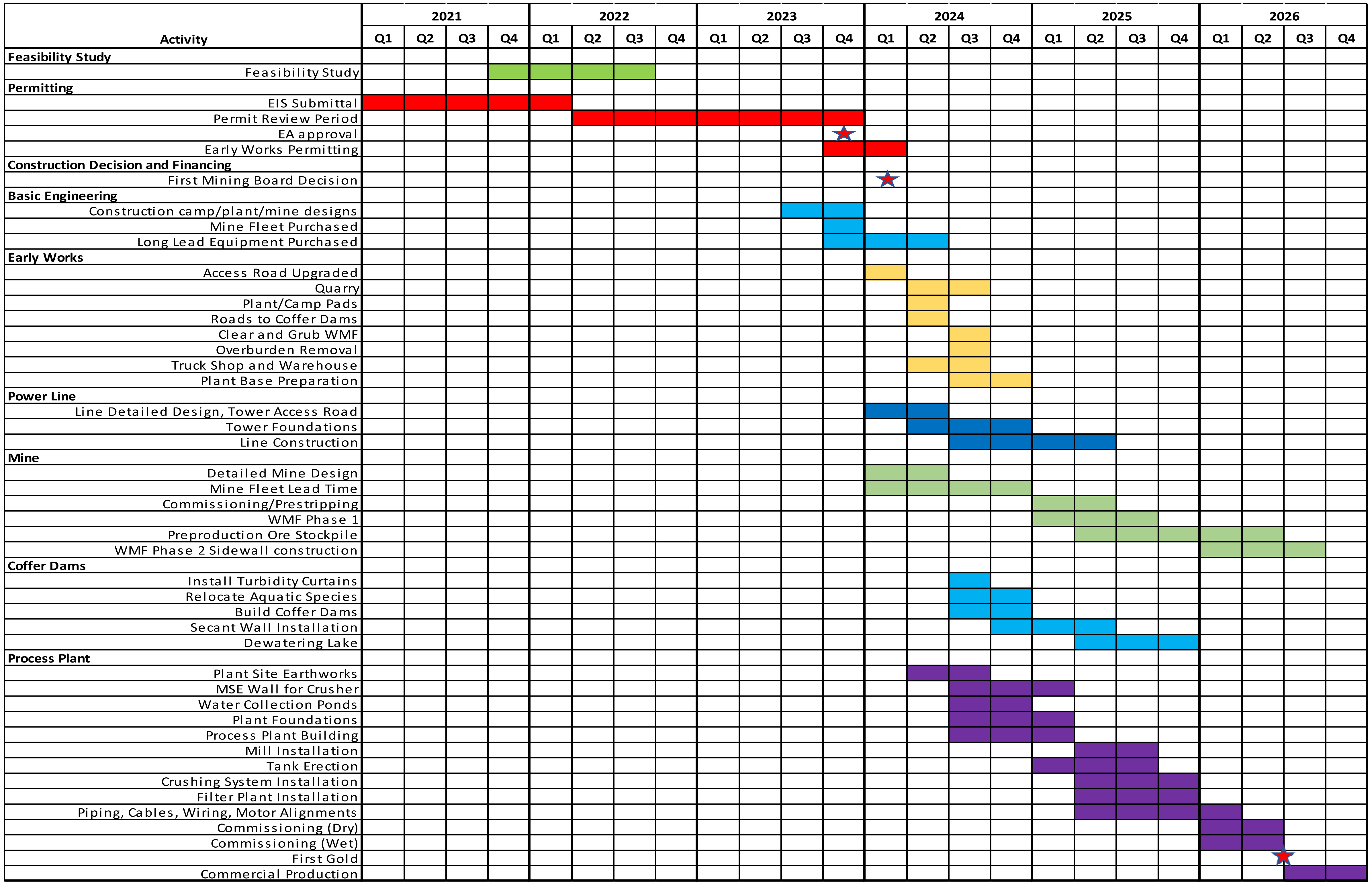
In a remote project with the scale of the Springpole Gold Project, proper scheduling to understand critical path items is paramount for project success. A PFS level project execution schedule has been developed in some detail to guide First Mining and the various discipline teams as they advance the Project forward.

A simplistic Gantt style chart has been included in this section to provide an outline of the major activities on the path towards a potential mine and give background on the cost estimates that resulted.

The dates indicated are best estimates at this time of expected permitting timelines and other construction activities. These may slow or accelerate for various reasons and are indicative at the time of this Report.

The current project execution schedule is shown in Figure 24-1.

Figure 24-1: Springpole Project Execution Schedule



25 INTERPRETATION AND CONCLUSIONS

25.1 Quality Assurance and Quality Control

The quality assurance/quality control program instituted by Gold Canyon and First Mining and conducted by SGS is of a standard generally consistent with current industry practice. SRK acknowledges that the QA/QC procedures have evolved and much of what is presented in this Report was "catch up" work. Included in this "catch up" work was an extensive resample program completed in 2013 of pre-2003 core to address the lack of QA/QC documentation for that drilling. In that respect, Gold Canyon did well to bring the database, at least from 2007 onward, up to an acceptable industry standard. The principal exceptions lie with:

- blank analyses suggest intermittent contamination introduced at some stage of material storage or processing
- lack of standard reference materials for silver

The analysis for gold and silver confirms an acceptable degree of reproducibility of samples for gold and a very good degree of reproducibility of samples for silver.

There is no evidence of bias in either gold or silver as a function of grade and First Mining has now implemented its own QA/QC procedures for deciding which assay batches are acceptable or not and which samples need to be re-assayed because of failed QA/QC.

The drill hole database from 2003 through 2020 is of a standard acceptable for public reporting of Mineral Resources.

25.2 Mineral Resource Estimate

A systematic re-sampling of the available drill core stored on-site at the Springpole Gold Project enabled the reclassification of a portion of the East Extension zone into the Indicated Mineral Resource category without the need to carry out an additional, extensive drilling campaign. This re-sampling exercise was a systematic program that included certified blanks and certified gold and silver standards.

The current mineral resource estimate for the Project prepared by SRK considers 662 core boreholes drilled by previous owners of the property during the period of 2003 to 2013, and seven holes drilled by First Mining in 2016 and 2020. The resource estimation work was completed by Dr. Gilles Arseneau, Ph.D., P.Geo. (APEGBC #23474), an independent Qualified Person as this term is defined in NI 43-101. The effective date of the resource statement is July 30, 2020.

In the opinion of SRK, the resource evaluation reported herein is a reasonable representation of the global gold and silver resources found in the Project at the current level of sampling. The Mineral Resources have been estimated in conformity with the CIM Estimation of Mineral Resource and Mineral Reserves Best Practices guidelines and are reported in accordance with the 2014 CIM Definition Standards and, at a 0.3 g/t Au cut-off, include 151 Mt grading 0.94 g/t Au classified as Indicated Mineral Resources and 16 Mt grading 0.54 g/t Au classified as Inferred Mineral Resources.

The current Mineral Resource estimate for the Project (July 30, 2020) was based on an Ag price of USD$1,550/oz and a gold price of USD$20/oz. These are considered reasonable economic assumptions by SRK. To establish a reasonable prospect of economic extraction in an open pit context, the resources were defined within an optimized pit shell with pit walls set at 35° to 50° based on domains. Assumed processing recoveries of 88% for Au and 93% for Ag were used. Mining costs were estimated at CDN$2/t of total material, processing costs estimated at CDN$1.62/t and G&A costs estimated at CDN$1/t. A COG of 0.3 g/t Au was calculated and is considered to be an economically reasonable value corresponding with breakeven mining costs. Approximately 90% of the revenue for the proposed Project is derived from Au and 10% from Ag.

Industry standard mining, process design, construction methods and economic evaluation practices were used to assess the Project. In SRK's opinion, there is adequate geological and other pertinent data available to generate a PFS.

Based on current knowledge and assumptions, the results of this study show that the Project has positive economics and should be advanced to the next level of study, a Feasibility Study.

As with almost all mining ventures, there are a large number of risks and opportunities that can influence the outcome of the Project. Most of these risks and opportunities are based on a lack of scientific information (test results, drill results, etc.) or the lack of control over external drivers (metal prices, exchange rates, etc.).

Subsequent higher-level engineering studies are needed to further refine these risks and opportunities, identify new ones, and define mitigation or opportunity implementation plans.

While a significant amount of information is still required to do a complete assessment, at this point there do not appear to be any fatal flaws for the Project.

25.3 Open Pit Mining

The PFS mine plan is based on open pit mining. This pit will provide the open pit feed material necessary to maintain the process plant feed rate at 30,000 tpd while operational.

The Springpole pit will be a three phase design which provides 121.6 Mt of mill feed grading 0.97 g/t Au, and 5.23 g/t Ag. Waste from this pit will total 275.4 Mt for a strip ratio of 2.6 (waste:mill feed).

In addition to the pit, a quarry will be established near the plant location in the pre-production period. This quarry will be used to construct mine infrastructure including haul roads, cofferdams and meet site fill requirements for other infrastructure. The quarry will contain 12.1 Mt of material that is planned to be mined.

The mill feed cut-off used is 0.40 g/t Au. During the mine operation material would be stockpiled to optimize the plant feed grade and defer lower-grade material until later in the mine schedule. Three grade bins are used for the stockpiles including: low grade (0.40 - 0.60 g/t Au), medium grade (0.60 – 0.80 g/t Au) and high grade (+0.80 g/t Au).

The phases are scheduled to provide 30,000 tpd of feed to the mill over a 12 year mining life after three years of pre-production stripping. The first two years of pre-production stripping are construction related. The last three years of mining are stockpile reclaim. The pits are sequenced to minimize initial stripping and provide higher feed grades in the early years of the mine life which the stockpiling strategy accomplishes.

The pits will be built on 12 m benches with safety berm placement each 24 m. Inter-ramp angles will vary from 39 to 54° in rock depending upon the wall orientation. Overburden uses a 30° inter-ramp angle with 12 m between berms. Minimum mining widths of 35 - 40 m were maintained in the design with preferred bench widths of 60 m or more. Ramps are at maximum 10% gradient and vary in width from 27.1 m (single lane width) to 35.4 m (double lane width). They have been designed for a 226 t haulage truck.

The main fleet will consist of three 251 mm rotary drills, two 36 m^3 electric hydraulic shovels and one 23 m^3 front end loader. The truck fleet will total seventeen 240 t trucks at the peak of mining. This is due to the long hauls from the pit to the WSF as well as the backhaul of tailings material from the plant. 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.

In the pre-production years -3, and -2, 3.9 Mt will be mined within the quarry area. This mining will be with 91 t trucks, 6m^3 excavators and smaller track drills, more suited to this type of work, preparing the site for the larger more productive equipment. Year -1 is the start of major mining activity using the larger equipment when the bay dewatering has advanced sufficiently for mining and the site infrastructure (power lines, roads, etc.) are in place. The early phases will provide the highest grade to the mill early in the schedule. The open pit will be in operation until Year 9 followed by three years of stockpile reclaim to feed the plant. When the open pit is complete, the larger mining fleet will move to complete the quarry area, dumping the material into the open pit. This serves to cover the slopes in the pit for reclamation purposes and upon closure the quarry will become additional lake area.

Waste material from the pit will be stored in the WSF. NAG material will be used for the outer berms while PAG material will be co-mingled with filtered tails. The filtered tails will be backhauled from trucks returning from dropping material at the plant either as feed or placed in the stockpiles. As the WSF advances upwards, re-sloping of the sides will be occurring to allow for concurrent reclamation and reducing the visual impact of the facility. The majority of the waste material will be contained within the WSF (196.6 Mm^3), but a small portion of NAG material will be backfilled into Phase 2 near the end of the mine life. This reduces the overall haul length and helps in reclamation of the pit. A total of 9.8 Mm^3 will be backfilled into the pit.

The mine equipment fleet is anticipated to be financed to lower capital requirements. The LOM operating cost is estimated at CDN$2.75/t mined (USD$2.06/t mined). This includes equipment financing of CDN$0.35/t mined (USD$0.26/t mined).

Pre-production stripping costs of CDN$105.8 M (USD$79.4 M) are capitalized. Other initial mine capital including equipment is CDN$86.8 M (USD$65.1 M). Sustaining capital is CDN$68.4 M (USD$51.3 M).

25.4 Mineral Processing and Metallurgical Testing

During 2020, First Mining completed a comprehensive comminution and metallurgical testwork program to support the PFS. This included head grade analyses, mineralogy, a full suite of comminution, flotation, and leach tests; cyanide detoxification, rheology, and solid/liquid separation.

Tests were performed on mineralization that is considered to be representative of plant feed, based on a recent mine plan. Composite samples representing major lithologies and a range of head grades were prepared. Bulk mineralogy on select composites showed the main sulphide mineral was pyrite with traces of chalcopyrite, sphalerite, and galena. A host of telluride minerals exist in the microscopic size range, with petzite the most dominant.

Comminution testing showed that the materials tested are considered very soft to medium in competency. Conventional Bond tests showed significant variation in hardness.

Two parallel flowsheets were evaluated, following the results from the previous studies: flotation + concentrate and tailings leaching versus whole ore leaching. The recommended flowsheet for this study is flotation with concentrate/tailings leaching.

Rougher flotation tests showed high sulphide recovery was generally achieved within eight minutes. Excessive foaming was observed in some samples. This was considered attributed to a drilling compound added to the core, to aid core recovery. High mass pull was observed in these samples. A cleaning stage reduced the mass pull reporting to concentrate regrind. Leaching of flotation tails is required to attain acceptable gold recovery. Tailings samples showed very high leach extractions in general.

Flotation concentrate gold extraction showed significant benefit from finer regrinding to an 80% passing size of 15 to 17 µm. Flotation concentrate gold extractions ranged from 62 to 97%, somewhat dependent on gold head grade. Flotation tails gold extractions ranged from 52 to 94%.

Overall plant gold recoveries are predicted to average 86% for head grades of 0.8 to 1.22 g/t Au. Overall plant recoveries for Ag are predicted to range from 85 to 92% for head grades of 3.2 to 8.3 g/t Ag.

Thickening and filtration of cyanide detoxified slurry showed a moisture content of 15% was achieved when employing a membrane squeeze in addition to pressing and drying in a plate and frame filter.

25.5 Infrastructure and Site Layout

25.5.1 Infrastructure

Key Project infrastructure as envisaged in the PFS includes: open pit mine area including mine haul roads and ramps; cofferdams for hydraulic isolation of the mine pit following bay dewatering; site main access roads, administrative access roads and maintenance roads, site main gate and guard house; administration and dry building, construction and permanent camp accommodation; process plant e-room; crushing area e-room; control room; reagent storage building; gold room; assay laboratory and sample preparation area; plant workshop and warehouse; truck shop and warehouse, tire changing facility, truck wash building; fuel facility, fuel storage and dispensing; fresh water intake; 230 kV overland and 25 kV power distribution lines; fresh water intake pumping supply and treatment; WSF, contact water collection ponds; waste water treatment plant and explosives magazine.

The main access road will be a private extension of the existing Wenesaga Road for forestry services and has been constructed up to 15 kilometres from the Project site.

Approximately 55MW of electrical demand will be supplied via a new 230 kV overhead transmission line, built to connect to the provincial grid's 230 kV line approximately 75 km to the southeast of the proposed mine. A 230 kV/25 kV transformer will provide step down prior to feeding a total of six electrical rooms. Variable frequency drives have been allowed where required and all medium-voltage motors or drives will be supplied in 4.16 kV.

25.5.2 Cofferdam

Two cofferdams will be constructed across Springpole Lake to isolate the open pit during operations (dewatering and mining).

The cofferdams will be constructed by controlled end-dumping of rockfill sourced from pre-production phase open pit stripping. Rockfill dumping will advance from the shoreline along the proposed centreline to construct a 28 m wide fill surface. The combined length of the two cofferdams will be 940 m. The maximum constructed dam height will be approximately 17 m.

The cofferdams will consist of a zoned embankment shell and a hydraulic barrier comprised of a Secant Pile Wall (SPW) and a grout curtain. The central zone of the embankment will be constructed with a 2" minus material to support installation of the SPW, while the outer zone of the embankment will be constructed with run of mine rockfill.

The SPW will isolate the open pit area from Springpole Lake and limit seepage through the cofferdams. The grout curtain is required to minimize seepage through the bedrock and limit pore pressure at the downstream toe of the embankment.

The closure and reclamation objectives for the Project will include returning the dewatered region to Springpole Lake through construction of a spillway within the cofferdams to allow natural refilling of the basin.

The design of the cofferdams is supported by geotechnical site investigations completed in 2018. Recommendations for additional supplemental geotechnical and hydrogeological site investigations are included in Section 26.

25.5.3 Waste Storage Facility

The WSF will store approximately 76 Mm^3 of tailings and 41 Mm^3 of PAG waste rock generated over the mine life. The WSF perimeter embankment is primarily constructed using NAG waste rock generated from open pit mining. A total of approximately 68.5 Mm^3 of fill material is required for construction of the embankment. The embankment will be constructed with 3H:1V upstream slopes and 2H:1V downstream slopes.

The WSF will be constructed in stages. Downstream raises of the perimeter embankment will be completed every two years until Year 6 to a final crest elevation of 470 m. The maximum embankment height is approximately 70 m.

Contact water (surface water run-off and water collected from the drainage system) will be pumped to a lined contact water management pond located south of the WSF.

WSF closure and reclamation works will commence in Year 13 with the aim of transitioning the facility into a self-sustaining landform. An engineered cover will limit infiltration when constructed. Foundation drainage will remain in place to monitor for and facilitate removal of water collected at the base of the facility following construction of the closure cover.

Limited geotechnical information was available to support the design of the WSF. It is assumed all overburden materials will be removed beneath the embankment foundations prior to fill placement. Recommendations for geotechnical and hydrogeological site investigations are included in Section 26.

25.6 Economic Analysis

The PFS is based on Probable Mineral Reserves that were converted from Indicated Mineral Resources.

The economic analysis has been run on a constant dollar basis with no inflation and the results converted to United States dollars for commonality in reporting against other projects. The Project has been evaluated on an after-tax basis.

Base case metal prices of USD$1,600/oz Au and USD$20/oz Ag are based on consensus analyst estimates and recently published economic studies. The forecasts are meant to reflect the average metal price expectation over the life of the Project.

The economic analysis was performed using the following assumptions:

- mine life (LOM) of 11.3 years
- exchange rate of USD$0.75 per CDN$1.00

- 100% ownership with 1.3% NSR (assumes buy back of 1.4% NSR)
- capital costs funded with 100% equity (no financing costs assumed)

The economic analysis was performed assuming a 5% discount rate. The pre-tax NPV discounted at 5% is USD$1,482 M; the internal rate of return (IRR) is 36.4%; and payback period is 2.2 years. On an after-tax basis, the NPV discounted at 5% is USD$995 M; the IRR is 29.4%; and the payback period is 2.4 years.

The cash flow model has been based on a three-year project development period, and that production commences in month one of Year 1.

Provision has been made for federal and Ontario taxation. The Canadian corporate income tax system consists of 15% federal income tax and 10% provincial income tax. Ontario applies a mining tax rate of 10%. Total undiscounted tax payments are estimated to be USD$720 M over the life of mine.

The Project has an outstanding NSR royalty of 3%, but First Mining can buy back a significant portion of this amount prior to production. The financial model assumes the exercise of this option, which would result in the financial model carrying a 1.3% NSR. The financial model assumes USD$5.3 M consideration associated with buyback of royalty.

Currently, the Project has a streaming agreement 50% of the payable silver produced over the life of the Project, but First Mining can buy back half of this amount prior to production. The financial model assumes the exercise of this option for USD$22.5M prior to production resulting in the model carrying 25% of the payable Ag stream.

The Project is most sensitive to revenue attributes such as gold price, head grade and exchange rate, followed by operating cost and capital cost.

25.7 Risks and Opportunities

The Springpole Gold Project has been examined by all disciplines to determine potential risks and understand mitigating measures that may be required to offset these. As well, opportunities exist that need to be highlighted for potential improvements in the overall project. Some examples of those risks are shown in Table 25-1 and the opportunities in Table 25-2.

Table 25-1: Springpole Project - Risks

Project Risk Area	Description	Possible Outcome	Mitigation
Mining	SW Pit Wall Slope	Flatter slopes may be required due to poor rock quality	Additional geotechnical drilling to accurately estimate expected slopes
	PAG/NAG Characterization	Increased PAG material could result in a larger WSF height and reduced NAG material for berm construction	Additional geochemical testwork to improve material understanding
	Water Inflows	Additional pumping requirements and site discharge to environment	Additional hydrogeological drilling/modelling to better understand water inflows
Mineral Processing	Tailings Filtration	Increased capital in filtration plant to generate filtered tailings that can be handled with haul trucks	Additional filter test work to properly size the filter plant for the various plant feed types
Waste Storage Facility	Foundation conditions are worse than expected	Increased overburden removal costs, material not suitable for an optional liner placement resulting in higher costs	Detailed site investigation of WSF site to properly categorize
Cofferdam	Foundation conditions worse than expected	May require additional cost to prepare foundation or ensure leakage is not above design	Additional site investigations

Table 25-2: Springpole Project – Opportunities

Project Opportunity Area	Description	Possible Outcome	Achieved
Geology	Inferred Material	Conversion of current waste material to Indicated and possible mill feed	Infill drilling to assist in resource classification
Mining	SW Pit Wall Slope	Steeper slopes which will help reduce waste movement	Additional geotechnical drilling to accurately estimate expected slopes
	PAG/NAG Characterization	Reduced PAG material could result in a smaller WSF height	Additional geochemical testwork to improve material understanding
	PAG material stored in Pit backfill	Reduced size of WSF Potential improved mine closure situation	Geochemical testwork to mimic subaqueous disposal for permitting
Mineral Processing	Tailings Filtration	Decreased plant capital in filtration	Additional filter test work to properly size the filter plant for the various plant feed types
Waste Storage Facility	Foundation conditions are better than expected	Reduced overburden removal costs, reduced quarry material requirements	Detailed site investigation of WSF site to properly categorize
Cofferdam	Foundation conditions better than expected	Reduced time for construction improving overall project schedule	Additional site investigations

26 RECOMMENDATIONS

26.1 Introduction

The PFS indicated a positive project. AGP recommends that First Mining proceed forward with additional studies including an FS. The recommendations and associated budgets by area are described further in the sub-sections.

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	$922,000
Geotechnical	$400,000
Metallurgy	$1,200,000
Infrastructure	$2,000,000
Environmental	$800,000
Feasibility Study	$3,000,000
TOTAL	**$8,322,000**

26.2 Geology

26.2.1 Inferred Resource Upgrade

In various pit areas it was noted that with a few drill holes, some of the material currently classed as Inferred Mineral Resources could be converted to Indicated Mineral Resources. This could result in material currently destined for the waste dump being processed. The bulk of the Inferred material is in the southern part of the pit, and approximately 4,100 m of drilling over 16 holes would be required to upgrade the material in this area to Indicated category.

26.2.2 Density Measurements

As part of the PFS work, an additional 471 core samples underwent bulk density testing. While sufficient for a PFS study, additional bulk density testwork is recommended to further improve the density model. This could be completed on existing core kept in storage on site.

26.2.3 Geology Estimated Budget

The following is the estimated budget for the proposed drilling program and other study work for the continued development of the mineral resources. The estimated budget for these proposed programs would be CDN$922,000 as shown in Table 26-2.

Table 26-2: Estimated Budget of Proposed Work

Proposed Work		Approximate Cost (CDN$)
Inferred Material Upgrade		
Diamond Drilling (~4,100 m)	$200/m	$820,000
Bulk Density Measurement; 500 samples	$35/sample	$17,500
Subtotal		$837,500
Contingency		$84,500
TOTAL		**$ 922,000**

26.3 Geotechnical

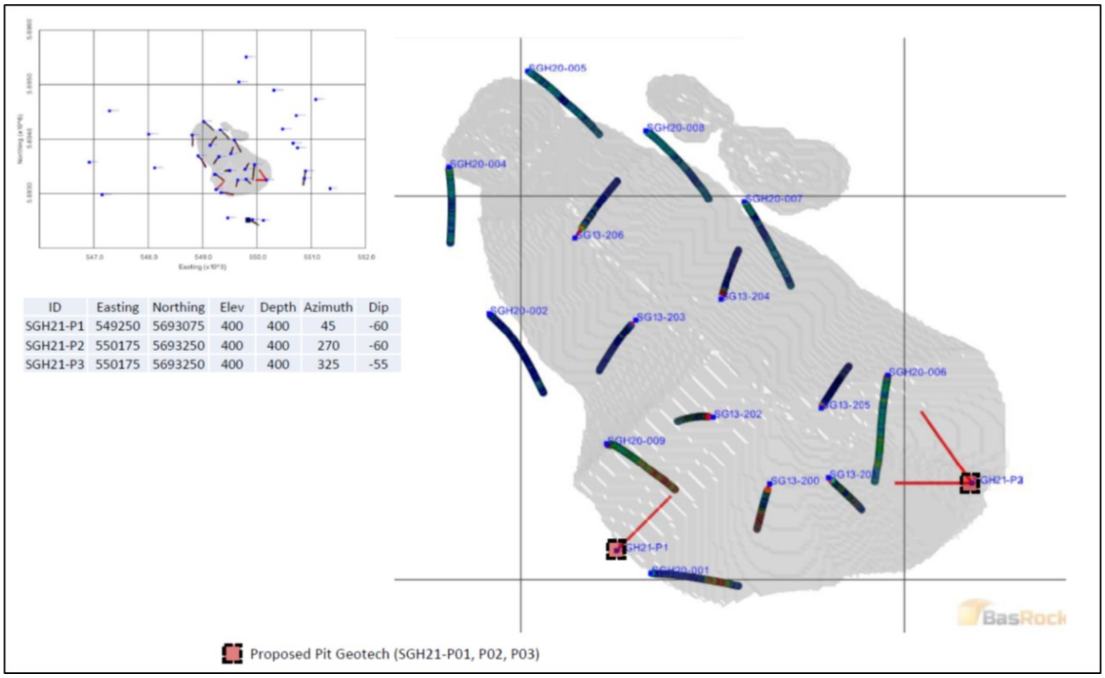
The results of the geotechnical gap analysis indicate a number of important factors that require additional investigation. For Feasibility designs, a higher level of confidence is required, and the preliminary geotechnical model presented will need to be updated with additional data and perhaps more importantly, additional integrated structural geotechnical interpretation and analyses. Recommended data collection and interpretation tasks are outlined below. These recommendations could potentially be completed in phases, using combinations of AGP, First Mining staff, and/or other contractors.

The following geotechnical work is recommended to advance the Project to an FS level:

Geotechnical Drilling and Core Logging

- A 3-hole geotechnical drilling and rock mass characterization program (Figure 26-1) is proposed to achieve FS data requirements, including targeted drilling of current data voids and areas of interest in the SW-S-SE sectors of the pit, to include discontinuity orientation measurements (where possible), sampling for laboratory strength testing, and televiewer surveys. The holes mainly target waste rock zones outside of the ore zone to determine the geotechnical properties of the units forming the pit walls. Planned metallurgy and infill holes in other areas of the pit could be used as dual purpose holes for collecting additional "quick / basic" geotechnical data. The core holes should be drilled using a triple tube core barrel to preserve the integrity of the core while drilling and retrieving.

Figure 26-1: Proposed Geotechnical Drillholes



ID	Easting	Northing	Elev	Depth	Azimuth	Dip
SGH21-P1	549250	5693075	400	400	45	-60
SGH21-P2	550175	5693250	400	400	270	-60
SGH21-P3	550175	5693250	400	400	325	-55

Source: AGP, 2021

- Laboratory testing is also recommended, including uniaxial compressive strength (UCS) testing (with strain measurements), triaxial strength testing, direct shear testing of discontinuities, and index testing of discontinuity infill materials. Results may be used to confirm or update the geotechnical analysis and slope design parameters provided. Samples should be collected from dedicated (or first-priority) geotechnical drill holes to ensure the appropriate materials are sampled, and to avoid conflicts with exploration sampling and assaying requirements. UCS and triaxial testing should be completed for each of the significant lithological units. The triaxial testing should focus on characterizing the intact rock strength both across and parallel to foliation. Samples of fault or dike contact gouge should also be collected and tested to help characterize the strength of these materials.
- Include the preliminary 3D lithostructural interpretation of all major faults, as viewed in drill core and rock outcrop within 200 m of the pit crest, and integrate them with the regional structural interpretation, into an exploration and geotechnical model for the FS.
- Produce robust 3D digital wireframe models of lithology, alteration with intensity, and structures.
- Update / confirm 2D and 3D stability models and assessments with the aim of optimizing pit slope designs according to anticipated geotechnical performance characteristics.

The following office-based data evaluation tasks are recommended to advance the Project to an FS level:

- Updating the existing 3D lithological and/or structural models to incorporate the results of all recent hydro-geotechnical drilling and/or an improved understanding of the deposit geology.
- Interpretation of structural and geotechnical data and development of a site geologic structural model incorporating major fault and shear structures.
- Consider transient numerical analyses, which take into account the actual mine sequencing.

These studies are estimated to cost a total of CDN$400,000 with CDN$250,000 for drilling and CDN$150,000 for studies, modelling and laboratory work.

26.4 Open Pit Mining

Building on the knowledge and the work completed in the PFS, the following is recommended for advancing the open pit design work to a Feasibility level of study:

1. Blasting Study
   a. Further evaluation of pattern sizes and powder factors is required to enhance production and productivity
2. Equipment Costs and Fleet Selection
   a. Update the equipment operating and capital costs from vendors
   b. Examine alternate vendors for specific equipment
   c. Optimize the size of haulage trucks
      - current fleet is 17 trucks of 240 tonne class
      - further study needed to determine most cost effective size
   a. Examine truck box configurations for tailings haulage to avoid carry back while maximizing carrying capacity of lower density tailings
3. Review if autonomous trucks are an opportunity
   a. Further study recommended
   b. Technical and social benefits concerns need to be examined
4. Ore Sampling Protocols need to be established
   a. Definition of ore/waste contacts
   b. Sample size selection
   c. Determine if blasthole samples can supplement the proposed Reverse Circulation (RC) samples
5. Haul Road Design
   a. Detailed design of north end haul road
   b. Detailed layout of haul roads for pre-production period
6. Pit Electrification Optimization
   a. 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 Feasibility study.

26.5 Mineral Processing and Metallurgical Testing

Additional metallurgical testwork should be conducted as part of future studies of the Springpole Gold Project. This testwork should:

- Include diamond drilling done using drill mud additives which assist with core recovery that have been demonstrated to have minimal impact on metallurgical testwork. A bulk sample might be considered to avoid the issue of drilling compound modifying reagents.
- Investigate the impact of drilling mud additives on flotation mass pull with the objective of reducing flotation circuit size and regrind power requirements.
- Further optimize concentrate leach reagents and consider reductions in leach extraction time. This includes reducing the number of concentrate leach adsorption tanks and recover residual Au/Ag in solution using the flotation tails CIP circuit.
- Optimize combined tails residual cyanide levels and aim to reduce cyanide detoxification retention time.
- Conduct a full Feasibility Study metallurgical testwork program incorporating variability and production composite testwork. This includes dewatering/filtering tests on the final tailings material.

These studies will require the generation of sufficient tailings to complete the filter testwork. To do this some pilot plant work may be required. The estimated cost is expected to range between CDN$800,000 and CDN$1,200,000.

26.6 Infrastructure

26.6.1 Site Layout

- Optimize the site layout to minimize quarry material requirements.
- Detail site water handling system with updated hydrogeological and surficial water studies.
- Re-examine the quarry for opportunities to maximize environmental benefit while reducing costs or enhancing closure benefits.

26.6.2 Cofferdam

- Continue site-specific meteorological and hydrology data collection to support refinement of seasonal run-off and design storm estimates. Additional evaluation of long-term regional meteorological data is recommended.
- A model of the Springpole Lake watershed behaviour under baseline (pre-mine) conditions is recommended to support evaluation of lake level fluctuation resulting from dewatering of the open pit area following cofferdam construction, along with lake level fluctuation resulting from design storm events. A baseline watershed model is expected to require monitoring of additional lakes within the Springpole Lake watershed, upstream of the lake outlet.
- Confirm foundation design parameters with additional site investigation, specifically focused on identification of any additional areas of soft lakebed sediments (clay, silt) within the footprint of the cofferdams. Investigation should target additional locations along the alignment of the

cofferdams, as well as the upstream and downstream toe of the embankments. Strength parameters must be determined to understand how the lakebed sediments and glacial till will behave under the embankment load (characterize material consolidation and expected strength gain under long-term loading).

- Additional hydrogeological characterization of the bedrock beneath the proposed cofferdams is recommended to support refinement of the extent and depth of bedrock grouting.
- It is recommended that samples of open pit waste and/or samples from potential borrow areas for cofferdam construction be collected for characterization of the rockfill material to be used for construction. Evaluation of the shear strength of the rockfill will confirm design parameters used in stability modelling. Geochemical classification of the proposed material source must be completed to confirm that the rockfill is NAG.
- Additional trade-off of alternative cut-off wall construction methods is recommended, following additional site investigation to evaluate a detailed bedrock profile.
- Additional development of a construction schedule is recommended as the design is advanced.

26.6.3 Waste Storage Facility

- Additional drilling and sampling of the overburden materials is recommended in the area of the WSF to better define the composition and expected variability in material depths. Geotechnical and hydrogeological drill holes are recommended at the maximum embankment sections, along with test pits within both the embankment and basin footprint. Laboratory testing should be completed to refine characterization of the material, including strength and hydraulic parameters for use in stability and seepage analyses.
- Determination of the filtered tailings critical state line is recommended to support selection of an appropriate undrained shear strength ratio for the material (or to support modelling as a drained material).
- It is understood that a NAG filtered tailings could be produced through removal of a cleaner tailings stream from the filtration circuit. It is recommended that concepts be developed for sub-aqueous storage of the PAG cleaner tailings to improve the overall NP of the WSF.
- Evaluation of the expected chemistry of the surface water run-off from the filtered tailings and PAG waste rock surface and water quality predictions in the water management pond downstream of the WSF will be important for site water management planning (i.e. determination of the requirement for water treatment prior to discharge, if the overall site is in a water surplus).
- Continue site-specific meteorological and hydrology data collection to support refinement of seasonal run-off and design storm estimates. Additional evaluation of long-term meteorological data is recommended.
- A long-term synthetic climate (temperature, precipitation, and evaporation) record for the site will support evaluation of the WSF water balance (along with water quality) based on historic dry and wet periods for the Project area, along with refinement of the overall site water management plan.

The proposed work is expected to cost in the order of:

- ongoing metallurgical monitoring work -CDN$ 500,000
- hydrometeorological reporting -CDN$50,000
- watershed modelling -CDN$100,000
- site investigation (including drilling) -CDN$1,000,000
- geochemical studies -CDN$250,000
- interface/overlap with EA/permitting -CDN$100,000

The sum of the studies is expected to cost CDN$2 M to complete.

26.7 Environmental

Building on the knowledge and the work completed in the PFS, First Mining will move forward with the concurrent federal and provincial EA process. The available environmental data will support this submission, but there are opportunities to add value to the Project generally, and the EA in particular as listed below:

- Store all project data in an appropriate database.
- Undertake additional data collection where earlier data may be becoming stale and review the historical data in the context of the latest project layouts. Updates as necessary for inclusion in the EA.
- Complete a water quality baseline report, based on data collection to date. This information should be used to provide source terms for water balance and watershed modelling.
- Commence continuous snow depth measurements at the Springpole climate station, as well as at least one season of winter snow surveying. Plans for ongoing snow surveying should be incorporated into the Project schedule.
- Commence measurement of precipitation as snowfall at the Springpole climate station using an all weather precipitation gauge, or heated rain gauge. Either installation should include a wind screen.
- Continue hydrometric monitoring on creeks around the Project area.
- Commence or continue lake level monitoring in nearby lakes potentially impacting or impacted by the Project.
- Completion of a comprehensive hydrometeorological baseline analysis that incorporates both project and regional surface water run-off and climate. This report should serve as a primary source for both mine planning (as described in earlier sections), water balance modelling, as well as all water related analyses and reporting associated with the EA.
- Complete a comprehensive assessment of the hydrogeological and hydrogeochemical conditions using collected data, with allocation for additional data collection for areas of increased uncertainty.
- Investigate the presence (or absence) of wolverine denning sites.
- Investigate the presence (or absence) of roost trees for Northern Myotes/Little Brown Myotis through a field reconnaissance of suitable habitat areas.
- Develop a stakeholder engagement database.

The sum of the recommended tasks is expected to cost CDN$800,000 to complete. This does not include compilation of the EA, but applies to studies that will support the EA.

26.8 Feasibility Study

The level of resource classification, and study work completed to date at the Springpole 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 Feasibility Study. This work by all the disciplines beyond the previous mentioned studies is estimated to be in the order of CDN$2-$3 M.

26.9 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 26-3: Summary of Recommendation Budgets

Area of Study	Approximate Cost (CDN$)
Geology	$922,000
Geotechnical	$400,000
Metallurgy	$1,200,000
Infrastructure	$2,000,000
Environmental	$800,000
Feasibility Study	$3,000,000
TOTAL	**$8,322,000**

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28 CERTIFICATE OF AUTHORS

28.1 Dr. Gilles Arseneau, Ph.D., P.Geo.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Technical Report and Pre-Feasibility Study on the Springpole Gold Project, Ontario, Canada" dated February 26, 2021, with an effective date of January 20, 2021 (the "Technical Report").

I, Dr. Gilles Arseneau, Ph.D., P.Geo., do hereby certify that:

1. I am an Associate Consultant with the firm SRK Consulting (Canada) Inc. ("SRK"), which has an office at Suite 2200-1066 West Hastings Street, Vancouver, British Columbia V6E 3X2, Canada.
2. I am a graduate with a B.Sc. in Geology from the University of New Brunswick in 1979; an M.Sc. in Geology from the University of Western Ontario in 1984 and a Ph.D. in Geology from the Colorado School of Mines in 1995. I have practiced my profession continuously since 1995. I have worked in exploration in North and South America and have extensive experience with Archean Gold deposits and porphyry hosted precious metal mineralization such as the Springpole Gold Project.
3. I am a Professional Geoscientist registered with the Association of Professional Engineers and Geoscientists of British Columbia, license # 23474.
4. I visited the Springpole Gold Project from February 10, 2012 to February 12, 2012 and from August 8, 2012 to August 9, 2012.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by virtue of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am independent of the issuer, First Mining Gold Corp., as defined in Section 1.5 of NI 43-101.
7. I am responsible for Sections 1.2 to 1.7, 1.9, 4 to 12, 14, 23, 25.1, 25.2, 26.2 of the Technical Report, as well as portions of Section 27, and accept professional responsibility for those sections of the Technical Report.
8. I have had prior involvement with the subject property contributing to the previous technical report for the Springpole Gold Project. I am a co-author of the previous technical report prepared in 2019 for the Springpole Gold Project.
9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Dated this 26th day of February, 2021, in Vancouver, British Columbia, Canada.

"signed and sealed"

Dr. Gilles Arseneau, Ph.D., P. Geo.
Associate Consultant
SRK Consulting (Canada) Inc.

28.2 Gordon Zurowski, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Technical Report and Pre-Feasibility Study on the Springpole Gold Project, Ontario, Canada" dated February 26, 2021, with an effective date of January 20, 2021 (the "Technical Report").

I, Gordon Zurowski, P.Eng., do hereby certify that:

- I am a Principal Mine Engineer with AGP Mining Consultants Inc., with a business address at #246-132K Commerce Park Dr., Barrie, Ontario L4N 0Z7, Canada.
- I am a graduate of the University of Saskatchewan with a B.Sc. 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.
- I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by virtue of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
- I am independent of the issuer, First Mining Gold Corp., as defined in Section 1.5 of NI 43-101.
- I am responsible for Sections 1.1, 1.10, 1.11, 1.13.1, 1.15 to 1.20, 2, 3, 15, 16.1, 16.2, 16.5 to 16.13, 18.1, 18.2, 18.5 to 18.9, 19, 21 (except 21.2.4 and 21.3.3), 22, 24, 25.2, 25.3, 25.5.1, 25.6, 25.7, 26.1, 26.4, 26.6.1, 26.7, 26.8, and 26.9 of the Technical Report and accept professional responsibility for those sections of the Technical Report.
- I have had no prior involvement with the Springpole Gold Project that is the subject of the Technical Report.
- My most recent site visit to the Springpole Gold Project was from July 13 to 15, 2020 for two days.
- As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.
- I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Dated this 26th day of February, 2021, in Stoufville, Ontario, Canada.

"signed and sealed"

Gordon Zurowski, P. Eng.

28.3 Roland Tosney, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Technical Report and Pre-Feasibility Study on the Springpole Gold Project, Ontario, Canada" dated February 26, 2021, with an effective date of January 20, 2021 (the "Technical Report").

I, Roland Tosney, P.Eng., do hereby certify that:

- I am a Senior Mining Geotechnical Engineer with AGP Mining Consultants Inc., with a business address at #246-132K Commerce Park Dr., Barrie, Ontario L4N 0Z7, Canada.
- I am a graduate of the University of Saskatchewan with a B.Sc. in Geological Engineering in 1998, and M.Sc. in Mining Geotechnical Engineering in 2001.
- I am a member in good standing of the Professional Engineers of Ontario (#100553567).
- I have practiced my profession in the mining industry continuously since graduation. My relevant experience includes over 20 years involvement in mining geotechnical assessments and feasibility studies in Canada, the United States, Central and South America, Europe, Asia, and Africa.
- I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by virtue of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
- I am independent of the issuer, First Mining Gold Corp., as defined in Section 1.5 of NI 43-101.
- I am responsible for Sections 16.3, 16.4, and 26.3 of the Technical Report and accept professional responsibility for those sections of the Technical Report.
- I have had no prior involvement with the Springpole Gold Project that is the subject of the Technical Report.
- I completed a site visit to the Springpole Gold Project from August 31 to September 1, 2020.
- As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.
- I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in accordance with that NI 43-101 and Form 43-101F1.

Dated this 26th day of February, 2021, in Saskatoon, Saskatchewan, Canada.

"signed and sealed"

Roland Tosney, P. Eng.

28.4 Cameron McCarthy, P.Eng., P.Geo., P.Tech.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Technical Report and Pre-Feasibility Study on the Springpole Gold Project, Ontario, Canada" dated February 26, 2021, with an effective date of January 20, 2021 (the "Technical Report").

I, Cameron McCarthy, P.Eng., P.Geo., P.Tech., do hereby certify that:

1. I am the President and Principal Engineer of Swiftwater Consulting Ltd., 22-1133 Ridgewood Drive, North Vancouver, British Columbia V7R 0A4 Canada.
2. I graduated with an undergraduate degree in Applied Science (Geology) from the University of Ballarat (Australia) in 2000. In addition, I have obtained a master's degree in Hydrotechnical Engineering from the University of British Columbia in 2013.
3. I am a member of Engineers and Geoscientists of British Columbia (EGBC), and the Association of Applied Technologists and Technicians of British Columbia (ASTTBC)
4. I have worked as a water resource engineer and environmental scientist for 20 years.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am independent of the issuer, First Mining Gold Corp., as defined in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Sections 1.14, 20, and 26 of the Technical Report and accept professional responsibility for those sections of the Technical Report.
8. I have had no prior involvement with the Springpole Gold Project that is the subject of the Technical Report.
9. I completed a site visit to the Springpole Gold Project from August 31 to September 1, 2020.
10. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.
11. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Dated this 26th day of February, 2021, in Vancouver, British Columbia, Canada.

"signed and sealed"
Cameron McCarthy, P.Eng., P.Geo., P.Tech.

28.5 Duke Reimer, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Technical Report and Pre-Feasibility Study on the Springpole Gold Project, Ontario, Canada" dated February 26, 2021, with an effective date of January 20, 2021 (the "Technical Report").

I, Duke Reimer, P.Eng., do hereby certify that:

1. I am employed as a Senior Engineer with Knight Piésold Ltd., with an office at Suite 1400 - 750 West Pender Street, Vancouver, British Columbia V6C 2T8, Canada.
2. I am a graduate of the University of British Columbia, with a Bachelor of Applied Science in Civil Engineering, 2011. I have practiced my profession continuously since 2011. My experience includes performing and overseeing geotechnical engineering design, tailings management and water management studies, environmental assessments, and monitoring of construction activities for mining projects.
3. I am a registered Professional Engineer in good standing with Engineers and Geoscientists of British Columbia in the area of Civil Engineering (No. 43270). I am also a registered Professional Engineer in good standing with Professional Engineers Ontario in the area of Civil Engineering (No. 100553772).
4. I have read the definition of "qualified person" set out in National Instrument 43-101 S*tandards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I am independent of the issuer, First Mining Gold Corp., as defined in Section 1.5 of NI 43-101.
6. I am responsible for the preparation of Sections 1.13.2, 1.13.3, 18.3, 18.4, 25.5.2, 25.5.3, 26.6.2, and 26.6.3 of the Technical Report and accept professional responsibility for those sections of the Technical Report.
7. I have had no prior involvement with the Springpole Gold Project that is the subject of the Technical Report.
8. I visited the Springpole Gold Project on August 30 to September 1, 2020.
9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Dated this 26th day of February, 2021, in Vancouver, British Columbia, Canada.

"signed and sealed"

Duke Reimer, P. Eng.

28.6 Dr. Adrian Dance, Ph.D., P. Eng.

CERTIFICATE OF QUALIFIED PERSON

To accompany the technical report entitled: "NI 43-101 Technical Report and Pre-Feasibility Study on the Springpole Gold Project, Ontario, Canada" dated February 26, 2021, with an effective date of January 20, 2021 (the "Technical Report").

I, Dr. Adrian Dance, Ph.D., P.Eng., do hereby certify that:

1. I am a Principal Consultant with the firm SRK Consulting (Canada) Inc., which has an office at Suite 2200 – 1055 West Hastings Street, Vancouver, British Columbia V6E 3X2 Canada.
2. I am a graduate of the University of British Columbia in 1987 where I obtained a Bachelor of Applied Science, and a graduate of the University of Queensland in 1992 where I obtained a Doctorate. I have practiced my profession continuously since 1992 including twenty years as a consultant and have experience working in a number of gold operations around the world.
3. I am a Professional Engineer registered with the Associated of Profession Engineers & Geoscientists of British Columbia, license # 37151.
4. I have not visited the Springpole Gold Project property.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by virtue of my education, affiliation to a professional associated, and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am independent of the issuer, First Mining Gold Corp., as defined in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Sections 1.8, 1.12, 13, 17, 21.2.4, 21.3.3, 25.4, and 26.5 of the Technical Report and accept professional responsibility for those sections of the Technical Report.
8. I have had prior involvement with the subject property contributing to the previous technical report for the Springpole Gold Project. I am a co-author of the previous technical report prepared in 2019 for the Springpole Gold Project.
9. As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Dated this 26th day of February, 2021 in Vancouver, British Columbia, Canada.

"signed and sealed"
Dr. Adrian Dance, Ph.D., P. Eng.
Principal Consultant
SRK Consulting (Canada) Inc.