Spanish Mountain Gold Project

Prefeasibility Study NI 43-101 Technical Report

Likely, British Columbia, Canada

Centred at 5,828,000 N and 603,000 E
(NAD 83)

Prepared for: Spanish Mountain Gold Ltd.
Prepared by: Moose Mountain Technical Services

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QUALIFIED PERSONS:
Bill Gilmour, P.Geo., Geology, Co-Founder of Discovery Consultants
Marc Jutras, P.Eng., Resource, Principal, Mineral Resources, Ginto Consulting Inc.
Marc Schulte, P.Eng., Mining, Moose Mountain Technical Services
Frank Grills, P.Eng., Principal Project Manager, Moose Mountain Technical Services
Malcolm Cameron, P.Eng., Electrical Engineer, MCA Engineering Limited
Paul Staples, P.Eng., VP & Global Practice Lead, Comminution, Ausenco Engineering Canada Inc.
Les Galbraith, P. Eng., Specialist, Associate, Knight Piésold Limited
Sam Billin, P.E., President/Founder Linkan Engineering
Andrea Samuels, P.Geo., Geochemistry Lead, pHase Geochemistry

IMPORTANT NOTICE

This report was prepared for Spanish Mount Gold Limited (SMG) by Moose Mountain Technical Services (MMTS), Discovery Consultants (Discovery) for geology; Ginto Consulting Inc., (Ginto) for resources; pHase Geochemistry (pHase) for geochemistry; Ausenco Engineering Canada Inc., (Ausenco), for metallurgy, processing, on-site infrastructure, on-site power distribution off-site infrastructure; and Knight Piésold (KP) for the Tailing Storage Facility, Overall Site Water Management and Environmental, MCA Engineering Limited (MCA) for the on-site main substation, main transmission powerline and partial on-site electrical distribution, Linkan Engineering (Linkan) for the Water Treatment (collectively the "Project Consultants"). This document is meant to be read as a whole. This document contains the expression of the professional opinion of the Project Consultants (MMTS, Discovery, Ginto, pHase, Ausenco, KP, MCA, and Linkan) based on (i) information available at the time of preparation, (ii) data supplied by outside sources, (iii) conclusions of other technical specialists named in this report, and (iv) the assumptions, conditions, and qualifications in this report. The quality of the information, conclusions, and estimates contained herein are based on industry standards for engineering and evaluation of a mineral project and are consistent with the intended level of accuracy for a Prefeasibility Study. This report is intended for use by SMG subject to terms and conditions of its contract with MMTS. Except for the purpose legislated under Canadian provincial and territorial securities law, any other uses of this report by a third party are at that party's sole risk.

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Units of Measure

Amperes	A
Canadian dollars	$ or C$
Centimetre cubic	cm^3
Centimetre	cm
Cubic feet per minute	cfm
Degree	°
Degrees Celsius	°C
Dry Tonnes	dmt
Feet	ft
Gallons per day	gpd
Gallons per minute	gpm
Gallons	gal
Gram	g
Grams per cubic centimetre	g/cm^3
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
Inches	"
Kilogram	kg
Kilograms per cubic metre	kg/m^3

Kilograms per square metre	kg/m^2
Kilometre square	km^2
Kilometre	km
Kilopascals	kPa
Kilovolt	kV
Kilowatt	kW
Less than	<
Litre	L
Litres per minute	L/min
Mega-annum (1 million years)	Ma
Megavolt ampere	MVA
Megavolt	MV
Megawatt	MW
Metre cubic	m^3
Metre level (relative metres level below surface)	m Level or mL
Metre square	m^2
Metre	m
Metres above sea level	masl
Metres cubic per hour	m^3/h
Micron	µm
Millimetre	mm
Million litres per day	ML/d
Million tonnes per annum	Mt/a
Million tonnes	Mt
Million years (annum)	Ma
Million	M
Ounce (troy ounce – 31.1035 grams)	oz
Ounce per annum	oz/a
Ounce per tonne	oz/t
Part per million	ppm
Percent by mass	%m
Percent mass fraction for percent mass	%w/w
Percent	%
Pound	lb
Pounds per square inch gage	psig
Tonnes per cubic metre	t/m^3
Tonnes per day	t/d
Tonnes per hour	t/h
United States dollars	US$
Volt	V

Abbreviations and Acronyms

Acid Desorption and Refining	ADR
Acid Rock Drainage	ARD
All-Terrain Vehicle	ATV
ALS Minerals Laboratory	ALS
American Association of Cost Engineers	AACE
American Smelting and Refining Company	Asarco
American Wire Gauge	AWG
Atomic Absorption	AA
Atomic Emission Spectroscopy	AES
Ausenco Engineering Canada Inc.	Ausenco
Ball Mill Work Index	BWi
BC Water Quality Guidelines	BCWQG
Best Achievable Technology	BAT
Biochemical Reactors	BCRs
Bond Abrasion	Ai
British Columbia Geological Survey	BCGS
Canadian Council of Ministers of the Environment	CCME
Canadian Development Expense	CDE
Canadian Exploration Expense	CEE
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
Capital Expenditure	CAPEX
Carbon-in-pulp	CIP
CDA Guidelines	CDA
Certified Reference Materials	CRM
Close-Circuit Television	CCTV
Coarse Ore Stockpile	COS
Coefficient of Variation	CV
Constituents of Concern	COCs
Copper	CU
Cumulative Probability Plots	CPP
Cumulative Tax Credit Account	CTCA
Cut-and-Fill	C&F
Cut-Off Grade	COG
Cyanide	CN
Diamond Drill Hole	DDH
Discovery Consultants	Discovery
Drop-Weight index	DWT
Effective Grinding Length	EGL

Electromagnetic	EM
Engineering-Procurement-Construction-Management	EPCM
Engineers and Geoscientists of British Columbia	EGBC
Environmental Design Flood	EDF
Environmental Management Act	EMA
Extended-Gravity Recoverable Gold	E-GRG
Feasibility Study	FS
Fire Assay	FA
Footwall	FW
Front-End-Loader	FEL
General and Administrative	G&A
Ginto Consulting Inc.	Ginto
Global Positioning System	GPS
Gold by Fire Assay	AuFA
Gravity Recoverable Gold	GRG
Gravity/Flotation/Leach	GFL
Half Absolute Relative Difference	HARD
Hangingwall	hw
Heating, Ventilation, and Air Conditioning	HVAC
High-Density Polyethylene	HDPE
High-Density Sludge	HDS
Hydrochloric acid	HCl
Independent Power Producer	IPP
Induced Polarization	IP
Inductively Coupled Plasma	ICP
Inductively-Coupled Plasma Atomic Emission Spectroscopy	ICP-AES
Inspectorate laboratories	Inspectorate
Intensive Leach Reactor	ILR
Internal Rate-of-Return	IRR
International Organization for Standardization	ISO
Inverse Distance Cubic	ID^3
Inverse Distance Square	ID^2
Inverse Distance Weighting	IDW
Inverse Distance	ID
Knight Piésold Ltd.	KP
Life-of-Mine	LOM
Light Detection and Ranging	LiDAR
Linkan Engineering	Linkan
Load-Haul-Dump	LHD

Long Lake Hydroelectric Project	LLHP
Low-Density Sludge	LDS
Mass Spectrometry	MS
MCA Engineering Limited	MCA
Microfiltration	MF
Mines Act Permit Application	MAPA
Mining-Influenced Water	MIW
Ministry of Energy, Northern Development and Mines	MENDM
Ministry of Natural Resources and Forestry	MNRF
Ministry of Natural Resources	MNR
Ministry of Northern Development and Mines	MNDM
Moose Mountain Technical Services	MMTS
Mountain Boy Minerals Inc.	MBM
National Instrument	NI
National Topographic System	NTS
Nearest Neighbour	NN
Net Present Value	NPV
Net Profit Interest	NPI
Net Smelter Return	NSR
Non-Potentially Acid Generating	NPAG
Operating Expense	OPEX
Ordinary Kriging	OK
Passive Treatment Systems	PTSs
pHase Geochemistry	pHase
Personal Computer Local / Wide Area Network	PC LAN/WAN
Potentially Acid Generating	PAG
Preliminary Economic Assessment	PEA
Preliminary Feasibility Study	PFS
Qualified Person	QP
Quality Assurance	QA
Quality Assurance/Quality Control	QA/QC
Quality Control	QC
Quality Management System	QMS
Reverse Circulation Drilling	RC
Reverse Osmosis	RO
Right-of-way	ROW
Rock Quality Designation	RQD
Rock Storage Facilities	RSFs
SAG Mill Comminution	SMC
SAG/Ball	SAB

Semi-Autogenous Grinding	SAG
Seepage Collection Pond	SCP
SGS Minerals Services	SGS
Sodium Cyanide	NaCN
Spanish Mountain Gold Project	the Project
Specific Gravity	SG
Standard Deviation	SD
Standard Operating Procedures	SOP
Sulfur Dioxide	SO_2
Sulfur	S
Sulfide Sulfur	S=
Tailings Storage Facility	TSF
Total Suspended Solids	TSS
Undepreciated Capital Cost	UCC
Underground	UG
Universal Transverse Mercator	UTM
Variable Frequency Drive	VFD
Very High Frequency	VHF
Voice over Internet Protocol	VoIP
Warning Level	WL
Water Treatment Plant	WTP
Weakly Acidic Dissociable CN	CN_{WAD} or WAD
Weight by Volume	w/v
Weight by Weight	w/w
Water Balance Model	WBM
Water Management Pond	WMP
Water Quality Model	WQM
Work Breakdown Structure	WBS
X-ray fluorescence spectrometry	XRF

1 SUMMARY

Spanish Mountain Gold Limited (SMG or the Company) is a Canadian-based exploration and development company based in Vancouver, Canada. Shares of SMG are currently traded on the Toronto Stock Exchange and OTC. SMG is focused on advancing its multi-million-ounce Spanish Mountain gold project in southern central British Columbia.

This Prefeasibility Study (PFS) is based on open pit mining operations feeding 20,000 tonnes per day (t/d) to the processing facility. The mining operation will be sequenced over a 14-year period Life-of Mine (LOM) to initially produce 2.1 million troy ounces (Moz) of gold and 0.9 Moz. of silver. The project benefits from existing road access, available grid power and BC Hydro substation connectivity, local supplies and labour availability, available water source/supply, and year-round access.

Access for production at the deposit will be through newly constructed haul roads. Mining methods will largely consist of conventional open pit methods. Ore will be trucked to the processing facility, and mining waste will be utilized to construct a new Tailing Storage Facility (TSF) and Water Management Pond (WMP), and the balance stockpiled on site at various Waste Rock Storage Facilities (WRSF) locations.

The processing plant including process and non-process facilities will be constructed within a construction period of approximately 24 months. The process plant was designed using conventional processing unit operations. It will treat 20,000 tonnes/day or 913 tonnes/hour based on an availability of 8,059 hours per annum or 92%. The crusher plant section design is set at 70% availability. The plant will operate with two shifts per day, 365 days per year, and will produce doré bars.

The Project will include a TSF and Water Treatment facilities WTF). The site will receive power via a 138 kilovolt (kV) power line from a new BC Hydro Substation constructed near Highway 97, to be known as SMMX Substation. A new receiving substation will be constructed adjacent to the processing plant that would receive power from a new 138kV transmission Line connecting to the new SMMX Substation. The transmission line will be owned and operated by SMG.

1.1 Introduction

To complete this Prefeasibility Study, SMG engaged a team of independent consultants, co-ordinated by Moose Mountain Technical Services (MMTS). The Technical Report has been prepared by MMTS in conjunction with Discovery Consultants (Discovery) for geology; Ginto Consulting Inc., (Ginto) for resources; pHase Geochemistry (pHase) for geochemistry; Ausenco Engineering Canada Inc., (Ausenco), for metallurgy, processing, on-site infrastructure, off-site infrastructure and on-site electrical distribution; and Knight Piésold (KP) for the Tailing Storage Facility, Overall Site Water Management, Water Balance and Environmental, Linkan Engineering(Linkan) for the Water Treatment; MCA for the main electrical transmission line and partial on-site electrical distribution;

and is written to comply with standards set out in National Instrument (NI) 43-101 for the Canadian Securities Administration (CSA).

1.2 Key Outcomes

The NI 43-101 highlights are as follows:

- Initial capital expenditure of $607.2M.
- 14-year Life-of-mine (LOM) sustaining capital of $290.5M, excluding closure capital.
- LOM operating costs of $19.38/t processed.
- Base case pre-tax net present value (NPV) 5% of $848M, internal rate of return (IRR) of 25%, and after-tax payback period of 3.2 years.
- Base case after-tax NPV5% of $655M and IRR 22%, and payback period of 3.3 years.
- Proven and Probable Mineral Reserves of 95.9 Mt, 0.76 g/t gold and 0.71 g/t silver (2.3 Moz. gold, 2.2 Moz. silver).
- LOM recovered production of 2.1 Moz. of gold and 0.9 Moz. of silver.

All currency amounts are referred to in Canadian dollars ($) unless otherwise indicated. All units of measure used in this report are metric or otherwise stated. All supporting documents cited in this report are referenced in Section 27.

1.3 Reliance on Other Experts

Non-QP specialists relied upon for specific advice includes:

- PricewaterhouseCoopers - post tax analysis of financial model, including Canadian Federal corporate income tax, BC Provincial income tax, and BC Mineral tax.

1.4 Property Description and Location

The Property is in the Cariboo region of central BC, 6 km east of the community of Likely, and 66 km northeast of the City of Williams Lake. The Property consists of 50 Mineral Titles Online (MTO) mineral claims, of which 20 are legacy claims. These mineral titles form a contiguous block covering an area of approximately 9,319 ha. The Property is 100% owned by SMG; subject to four separate net smelter return (NSR) royalties on some of the mineral tenures. Figure 1-1 shows the general location of the Project.

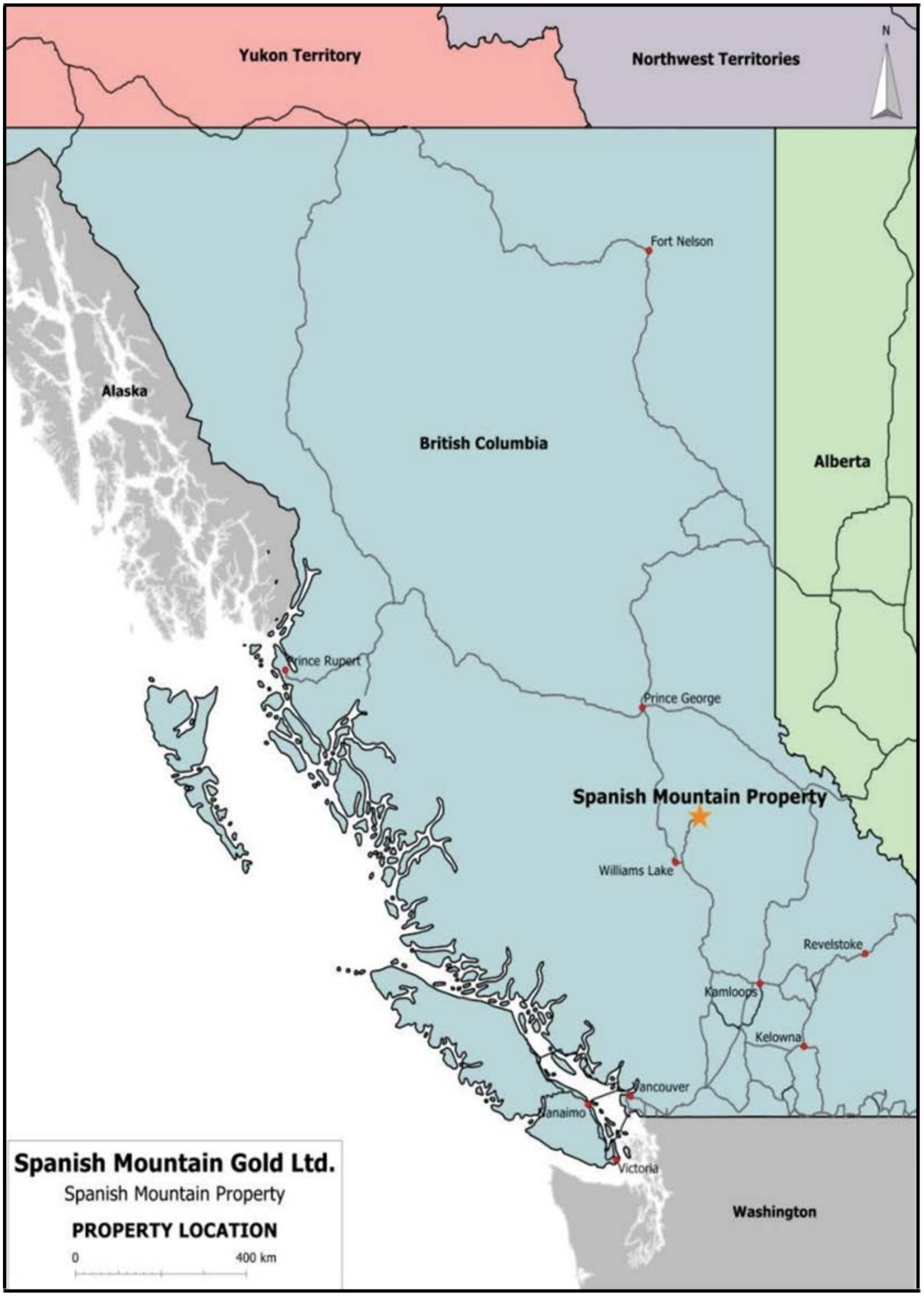


Figure 1-1 Project Location

The main resource, consisting of the Main and North Zones, is located west of the northwest end of Spanish Lake, and is centred at approximate Universal Transverse Mercator (UTM) coordinates 604,425 East and 5,827,900 North (NAD 83, Zone 10). It is located mainly within mineral claim 204667 and mineral claims 204225 and 204226.

The Property can be reached from Williams Lake via the Likely road, which is a paved secondary road that leaves Highway 97 at 150 Mile House, approximately 16 km south of Williams Lake, and continues for 87 km to Likely. From Likely, the Property is accessed from the Spanish Mountain 1300 Forest Service Road (FSR).

The Resource underlies areas on which royalties are payable, which may or may not significantly affect the economic potential of the Property. Other than these issues, to the extent known, there are no significant factors and risks that may affect access, title or right or ability to perform work on the Project.

1.5 Accessibility, Climate, Local Resources, Infrastructure, and Physiography

1.5.1 Accessibility

The Property can be reached from Williams Lake via a paved secondary road that leaves Highway 97 at 150 Mile House, approximately 16 km south of Williams Lake, and continues for 87 km to Likely (Figure 5-1). From Likely, the central and northern part of the Property is accessed from FSR 1300, which begins east of Likely and continues through the centre of the Property. The southern portion of the Property is accessed from Likely along the Cedar Creek / Winkley Creek Road (FSR 3900), for about 10 km. Numerous logging roads lie throughout the Property and offer good access to most areas. A gravel airstrip is located along the 1300 FSR between kilometres 2 and 3.

1.5.2 Climate

The climate of the Likely area is modified continental with cold snowy winters and warm summers. Likely has annual average precipitation of approximately 70 cm. Snowfall on the Property is commonly about 200 cm between the months of October and April. Most small drainages tend to dry up in the late summer. Drilling programs can be conducted on a year-round basis.

1.5.3 Local Resources and Infrastructure

The main access area to the Property is the Likely Road, which passes north of the access road to the Mount Polley copper-gold mine, owned by Imperial Metals Ltd. This mine is situated about 15 km southwest of the centre of the Property. Power is available at Likely, with a major line in place to Mount Polley. Water is abundant in the area.

SMG has a modern, full-service facility on purchased land near the Property that provides a base for operations. Likely has basic amenities including a motel, hotel, rental cabins, corner store, gas pumps, and a seasonal restaurant. Some heavy equipment is also available for hire from local contractors. All services and supplies are readily available in Williams Lake, an hour’s drive from Likely. The Williams Lake airport is serviced by Central Mountain Air and Pacific Coastal Airlines, which provide daily service with Vancouver, BC, and by Air Canada, which provides less frequent service.

1.5.4 Physiography

The Property covers an area of up to approximately 14 km north to south by 15 km east to west, situated between Spanish Lake on the east and Quesnel Lake on the west. Physiographically, the area is situated within the Quesnel Highland, which is transitional between the gently undulating topography of the Cariboo Plateau to the west, and the steeper, sub-alpine to alpine terrain of the Cariboo Mountains to the east. The terrain is moderately mountainous with rounded ridge tops and U-shaped valleys. Topography is locally rugged with occasional cliffs and moderately incised creek valleys. Within the Property, elevations range from 910 masl at Spanish Lake to 1,587 masl at the Peak of Spanish Mountain. Drainage is via Spanish Creek, which drains to the northwest into Cariboo Creek, and via Cedar Creek, which drains to the west into Quesnel Lake. Quesnel Lake flows into Quesnel River, and joined by Cariboo Creek, flows westerly to eventually join the Fraser River near the town of Quesnel.

Vegetation in the area consists of hemlock, balsam, cedar, fir, and cottonwood found in valley bottoms and spruce, with fir and pine at higher elevations. Alder, willow, and devil's club grow as part of the underbrush, which can be locally thick. Parts of the Property have been logged at various times, resulting in areas having open hillsides with younger forest growth. In addition, large sections of the pine forest have recently been affected by mountain pine beetle infestation.

There is sufficient land available within the mineral tenure held by SMG for tailings disposal, mine waste disposal, the process plant, and related site infrastructure.

1.6 History

The earlier history of the Property has been summarized by Page (2003), and by Singh (2008). Table 6-1 gives a summary of the historical work, up to and including 2004, in tabular form, and has been adapted from Singh (2008) with minor edits. The 2005 to 2009 exploration programs carried out by SMG at that time were done under its former name of Skygold Ventures Ltd. Work conducted from 2005 to the present is described in more detail in Sections 10 and 11 of this Report.

1.7 Geological Setting and Mineralization

1.7.1 Property Geology

Geologically, the SMG deposit is within metasediments of the Quesnel Terrane, and is hosted by the phyllite package of rocks, which comprises interbedded slaty to phyllitic, dark grey to black siltstone, carbonaceous mudstone, greywacke, tuff, and minor conglomerate. The main host of the gold mineralization is black, graphitic phyllitic argillite. The sedimentary units have been intruded by plagioclase-quartz-hornblende sills and dykes, which range in thickness from tens of centimetres to as much as 100 m. The intrusions have also been affected by phases of folding, alteration, and quartz veining.

1.7.2 Stratigraphy

The stratigraphy of the SMG deposit has been summarized by Singh (2008). Slightly revised, it comprises the following stratigraphic sequence from northeast to southwest, and stratigraphically higher to lower:

- **North Zone Argillite**: fine-grained, black argillite with siltstone interbeds, generally 30 to 100 m thick. Interbeds of altered tuff also occur. This unit hosts wide zones of disseminated gold mineralization. Alteration consists of ankerite, sericite, pyrite, silicification, and quartz veining.
- **Altered (Upper) Siltstone** (with mafic dykes): medium to light grey, finely laminated, up to 130 m thick. Several altered mafic dykes are present. Visible gold has been noted in quartz veins in several locations. Alteration consists of chromium-rich sericite, ankerite, silicification and quartz veining.
- **Main Zone (Upper) Argillite**: Black, graphitic, locally finely laminated. The unit is up to 100 m thick, with contorted bedding (cataclastic deformation) and is locally friable and faulted. Alteration consists of occasional ankerite and minor quartz veins. The bulk of the disseminated gold mineralization (>65%) is hosted in this unit.
- **Lower Tuff - Greywacke** (with mafic dykes): Often mottled, light to dark grey, fine to coarse-grained tuffs with lesser siltstones, greywackes, and minor felsic dykes. Local argillite horizons are also present. The unit is often strongly silicified, and sometimes pervasive alteration (sericite–ankerite–silica) has made identification of the original rock type very difficult. Visible gold is often found in quartz veins. It also contains thin sills of a probable mafic intrusion.
- **Conglomerate**: medium–grained, angular to sub-rounded, clast supported. Clasts are commonly siltstone, tuff, and greywacke. The unit is narrow (< 1 m), however, it is useful as a marker horizon at the base of the Lower Tuff – Greywacke sequences.
- **Lower Argillite** (with tuffs and siltstone): black to dark grey, interbedded argillite, tuff, and siltstone, with minor felsic dykes. This unit exhibits ankerite and silica alteration and only minor graphite. Pyrite content is generally less than 2%. The unit hosts lesser to minor amounts of gold mineralization.

1.7.3 Mineralization

Gold mineralization occurs as two main types:

1. Disseminated within the black, graphitic argillite. This is the most economically significant form. Gold grain size is typically less than 30 microns, and is often, but not always, associated with pyrite. Disseminated gold has also been associated with quartz veins within faults zones in the argillite.

2. Within quartz veins in the siltstone/tuff/greywacke sequences. It occurs as free, fine to coarse (visible) gold and can also be associated with sulphides including galena, chalcopyrite, and sphalerite. Highest grades have come from coarse gold within quartz veins.

Disseminated gold within the argillite units is by far the most potentially economically important type of mineralization, and has been traced for over 2 km, occurring in multiple stratigraphic horizons. From drill core, elevated gold content has been noted within fault zones as well as within quartz veins in fault zones. However, the influence of fault zones in relation to the gold content of the deposit is not certain.

In the opinion of the QP, William Gilmour, P.Geo. the regional setting and local geology are sufficiently well understood to support the estimation of Mineral Resources.

1.8 Deposit Types

Mineral deposits in the SMG gold deposit are classified as a sediment-hosted vein (SHV) deposit. In addition, the metal chemistry is gold without an association of other trace elements. There is also a lack of significant base metal sulphides.

1.9 Exploration

Exploration for the Main and North Zones is based on the results of sampling of both drill core and Reverse Circulation (RC) rock chip samples (cuttings) from the programs carried out from 2005 to 2018. Programs carried out before 2005 are summarized in Section 6 – Exploration History. Note that the 2005 to 2009 exploration programs carried out by SMG were done under its former name of Skygold Ventures Ltd. A more complete summary of the drilling programs from 2004 to 2020 is in Section 10.

The exploration programs completed to date are appropriate for the style of the mineralization and prospects located on the Project.

1.10 Drilling

In 2004, Wildrose Resources Ltd. carried out drilling on the Property. SMG has been drilling on the Property since 2005.

For the 2005, 2006 and 2007 core drill programs, drilling was contracted to LDS Diamond Drilling of Kamloops BC. The 2008 core drill program was contracted to North Star Drilling. In 2007, the main drill direction of 210 degrees was less than optimal, as it is sub-parallel to known regional faults. Drill directions were changed slightly, and consequently, this reduced the number of holes lost in fault zones.

The 2004 RC drill program was carried out by Northspan Exploration of Kelowna, BC. The 2005 and 2006 RC drilling was done by Drift Exploration Drilling of Alberta, and Northspan Exploration of Kelowna BC.

For the 2010, 2011, 2012 and 2018 programs, core drilling was contracted to Atlas Drilling Company of Kamloops, BC. Downhole measurements including azimuth and dip were measured using a Reflex EZ-Shot® tool and were collected every 50 m down hole. Collar locations were initially surveyed using a hand-held GPS. Once drilling was completed, the 2010 drill collar locations were more accurately surveyed by Crowfoot Surveys of Kamloops, BC, utilizing standard surveying equipment. Surveying in 2011 and 2012 was done in-house using Trimble R8R2K Survey® GPS equipment supplied by Cansel Survey Equipment Inc.

For the 2013, 2014, 2018 and 2020 programs, RC drilling was contracted to Northspan Explorations Ltd, of Kelowna, BC. Drilling was done using a skid-mounted Super Hornet drill utilizing five-foot drill rods. A 5.5-inch (140 mm) casing was run through the overburden into solid bedrock, followed by a 4.0-inch (102 mm) diameter drill bit for sample collection. A couple of holes were drilled with a 3.5-inch diameter bit. All samples below the casing represented five-foot (i.e., 1.52 m) sections of rock cuttings, equivalent to rod length.

There was no downhole surveying of RC drill holes. For core drilling, any downhole surveying is mentioned below where applicable. Drill recovery is generally good, although RC recovery appears to be better than core recovery.

The locations of the 2004 to 2020 drillholes are shown on Figure 10-1 through Figure 10-14. Representative examples of drill sections through the mineral deposit are shown in Figure 10-15 and Figure 10-16.

In the opinion of the QP the quantity and quality of the data collected in the completed drillhole programs are sufficient to support the Mineral Resource Estimation. There are no known drilling, sampling or recovery factors that could materially impact the accuracy and reliability of the results.

1.11 Sample Preparation, Analysis, and Security

Sampling methods were used by SMG in the 2010, 2011, 2012, 2018 and 2020 core drilling programs and in the 2013, 2014 and 2018 RC drilling program. Sampling methods are also used for the 2004 to 2009 programs. Section 11 presents information obtained from SMG, ALS Global Minerals Lab (ALS), and reports by co-author William Gilmour, P.Geo., who visited the Property on April 22, 2012, for the core drilling programs, on August 23, 2013, for the RC drilling program, and on September 12, 2019, and on December 2, 2020.

SMG has maintained a consistent program of independent assay QA/QC since 2005. The programs include the addition of blank samples, standards and pulp duplicates were inserted and analysed, together with repeat analysis. Control samples from the lab include control blanks, duplicates, and standards (for the 2010, 2011, 2012, 208 and 2020 core drilling programs. The sample blank was inserted at the beginning of the batch, standards were inserted at random intervals, and duplicates were analyzed at the end of the batch.

For the 2013, 2014 and 2018 RC drilling programs, similar QA/QC procedures were adopted. At ALS laboratory, quality control samples from the laboratory included analytical control blanks, pulp duplicates and standards. The analytical sample blank was inserted at the beginning of the batch, then every 40 samples. Two lab standards were inserted per 40 samples. Four lab standards were used for the metallic screen analysis and four other standards were used for the multi-elemental analysis. A pulp duplicate was done every 20 samples on the multi-element analysis.

It is the opinion of the QP, William Gilmour, and P.Geo. that the sample security, sample preparation and analytical procedures during the exploration programs followed accepted industry practice appropriate for the stage of mineral exploration undertaken and are NI 43-101 compliant.

1.12 Data Verification

Data verification was carried out for all the drilling programs stated in Section 11. The 2013, 2014 and 2018 RC drill and core drill programs were carried out by SMG under the supervision of Judy Stoeterau, P.Geo. of SMG. Qualified Persons from Discovery Consultants monitored the drilling, sampling, QC/QA procedures, reviewing analytical certificates throughout the drill program. The co-author William Gilmour, P.Geo. was responsible for reviewing the results, including QA/QC, and at no time directed the activities of SMG staff.

For the 2011 and later drill programs, co-author Gilmour obtained the analytical results directly from ALS. The data included CSV files and PDF files. There were no discrepancies noted between the two types of data files. By using the sample templates produced in the field, the sample numbers for specific intervals of drill core or cuttings were matched up with the sample numbers in the analytical reports. The same procedure was done for field blank samples, field duplicate samples and field standards, producing a compiled spreadsheet of the all the results.

An independent validation of the drillhole database was carried out by Ginto Consulting Inc prior to the estimation of the Mineral Resources for the 2021 PFS. In this exercise approximately 10% of the gold assays from the drillhole database were checked against the original assay certificates. Similarly, approximately 10% of the drillhole collar coordinates and downhole surveys from the drillhole database were checked against the drill logs. Overall, no significant errors were found and the drillhole database was deemed valid for the estimation of Mineral Resources.

The process of reviewing the data used in the Mineral Resource Estimate has been reviewed by QP Marc Jutras, P.Eng., M.A. Sc., and in his opinion sufficient verification checks have been undertaken on the drillhole database to provide confidence that the database is of sufficient quality to support the Mineral Resource Estimate.

1.13 Mineral Processing and Metallurgical Testwork

The Spanish Mountain Gold deposit has been the subject of several metallurgical test programs, the most recent of which was carried out at McClelland Laboratories in 2019. For the 2020 PFS, metallurgical test work conducted by SGS, G&T, Met-Solve, Knelson, and McClelland Laboratories was reviewed.

Tests were performed on mineralization that is representative of the 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 over the life of mine. The grade variability samples gold and silver grades ranged from 0.15-3.25 g/t Au and 0.5–4.4 g/t Ag.

Most of the metallurgical testwork was performed from samples which originate from the central part of the planned open pit (called the main zone).

Bulk mineralogy on select composites showed that pyrite was the main sulfide mineral present, representing between 0.5-2.5% by mass. Sphalerite and chalcopyrite were also present in relative order of abundance. Gold occurs as free gold associated with quartz veins and as attachments to and occlusions in pyrite.

Comminution testing showed that the materials tested are highly variable in competency. The breakage data showed the ore can be classified as competent and moderately hard, and moderately abrasive with SAG Mill Comminution (SMC) tests ranging from 26-51.7. Conventional Bond tests showed significant variation in hardness, with Bond rod mill work indices ranging 12.4-17.5 kWh/t and Bond ball mill work indices ranging from 10.9-16.7 kWh/t.

Rougher flotation tests showed high sulfide recovery was generally achieved within 8 minutes of flotation time. Flotation recoveries to cleaner concentrate ranged 80-92% for gold, 25-55% for silver.

Total organic carbon (“TOC”) occurs in high enough concentrations in the feed to negatively impact gold leach recoveries because of preg-robbing. TOC was successfully depressed in the cleaner flotation circuit using CMC. No other deleterious elements are present at levels to be cause for concern.

High leach recoveries were achieved when leach feed was reduced to <0.5% TOC and following regrind to a P80 of 22µm.

Overall plant recoveries for gold are predicted to range from 85–92% for head grades ranging from 0.6-1 g/t Au. Overall plant recoveries for silver are predicted to range from 38–42%.

Cyanide detoxification tests reduced WAD cyanide levels to 1.5 mg/L with moderate reagent consumption rates.

There were three main changes to the process flowsheet from the 2019 PEA, i.e.

- Primary, secondary, and tertiary crushers were replaced by a SABC grinding circuit
- Primary gravity concentration was included as part of the grinding circuit together with an intensive leach reactor
- Conventional flotation cells were replaced by DFR flotation technology.

1.14 Mineral Resource Estimate

The mineral resource estimate of the Spanish Mountain Gold Project's Prefeasibility Study (PFS) represents an update of the 2019 Preliminary Economic Study (PEA). Although there are no new drillholes added, the grade estimation strategy has been revised for the PFS. The geology model previously used for the PEA study, was retained as the geology model for the PFS as no new drill hole data was available. The exploratory data and variographic analyses, grade estimation, and mineral resource classification, were revisited for the PFS. In addition to grade estimates for Au and Ag, grade estimates for As, Ca, and total S were provided in this study.

A total of 110,762 gold 1.5m composites from 815 holes and 109,769 silver 1.5m composites from 797 holes located within the mineralized wireframes were utilized in the grade interpolation process. The main geologic control on mineralization was identified as related to lithologic units. Gold grade continuity was found trending at an azimuth of 120° with down-dip angles varying from 25° to 65° northeast. The ranges of gold grade continuity vary from 49m to 73m along strike, from 36m to 82m down dip, and from 16m to 74m across strike and dip. The estimation of gold and silver grades was carried out with an ordinary kriging technique on capped 1.5m composites. A search ellipsoid dimensioned to the second range of the variogram models for each mineralized units was utilized for the grade estimation process into an orthogonal block model of 15m (X) x 15m (Y) x 5m (Z) blocks. Various validation tests were performed to assess the quality of the grade estimates. The mineral resource was pit constrained to provide "reasonable prospects of economic extraction".

The mineral resource estimate has been classified as "Measured", "Indicated" and "Inferred" according to the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) "CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines" (May 2014). The following Table 1-1 shows the Mineral Resources.

Table 1-1 Mineral Resources at a 0.15 g/t Au Cut-Off Grade

Classification	Tonnage Tonnes	Average Grade		Metal Content	
		Au g/t	Ag g/t	Au oz	Ag oz
Measured	68,429,000	0.59	0.67	1,289,000	1,474,000
Indicated	225,724,000	0.47	0.73	3,418,000	5,298,000
Measured+Indicated	294,153,000	0.50	0.72	4,707,000	6,772,000
Inferred	18,343,000	0.63	0.76	372,000	448,000

Notes:

1. *Mineral resources' tonnage and ounces have been rounded to the nearest thousand.*
2. *The Inferred mineral resources have a lower level of confidence than that applying to Indicated mineral resources and must not be converted to mineral reserves. It is reasonably expected that most Inferred mineral resources could be upgraded to Indicated mineral resources with continued exploration.*
3. *The mineral resources are constrained within a pit optimized at a $1,600US$/oz gold price and $20US$/oz silver price.*
4. *Inclusive of Mineral Reserves - Effective February 3, 2021.*

1.15 Mineral Reserve Estimates

Proven and Probable Mineral Reserves have been modified from Measured and Indicated Mineral Resources at Spanish Mountain Gold and are summarized in Table 1-2. Inferred Mineral Resources are set to waste.

Open pits are based on the results of Pseudoflow sensitivity analysis, and then designed into detailed pit phases to develop pit reserves for mine production scheduling.

Factors that may affect the Mineral Reserve estimates include metal prices, changes in interpretations of mineralisation geometry and continuity of mineralisation zones, geotechnical and hydrogeological assumptions, ability of the mining operation to meet the annual production rate, process plant and mining recoveries, the ability to meet and maintain permitting and environmental licence conditions, and the ability to maintain the social licence to operate.

Table 1-2 Proven and Probable Mineral Reserves

Reserve Class	Mill Feed (Mt)	Mill Feed Gold Grade (g/t)	Contained Metal (Moz)	Mill Feed Silver Grade (g/t)	Contained Metal (Moz)
Proven	40.8	0.79	1.03	0.67	0.88
Probable	55.1	0.74	1.31	0.74	1.31
Total	**95.9**	**0.76**	**2.34**	**0.71**	**2.19**

Notes:

1. *The Mineral Reserve estimates were prepared by Marc Schulte, P.Eng. (who is also an independent Qualified Person), reported using the 2014 CIM Definition Standards, and have an effective date of March 31, 2021.*
2. *Mineral Reserves are mined tonnes and grade; the reference point is the mill feed at the primary crusher.*
3. *Mineral Reserves are reported at a cut-off grade of 0.30 g/t Au. Cut-off grade assumes US$1,500/oz. Au at a currency exchange rate of 0.76 US$ per C$; 99.8% payable gold; US$5.00/oz. offsite costs (refining and transport); and uses an 80% low grade metallurgical recovery. The cut off-grade covers processing costs of $6.50/t, site and administrative (G&A) costs of $2.50/t, coverage for project sustaining capital costs of $3.00/t, and a stockpile rehandle cost of $2.00/t.*
4. *Mined tonnes and grade are based on an SMU of 15 m x 15 m x 5 m, including additional estimates for mining loss (3%) and dilution between ore and waste zones (6.6%, 0.24 g/t Au, 0.60 g/t Ag).*
5. *Numbers have been rounded as required by reporting guidelines.*

1.16 Mining Methods

The Spanish Mountain deposit will be mined using a conventional open pit mining method. A PFS level mine operation design, 14-year open pit production schedule, and mining cost model have been developed.

The mining fleet will include diesel-powered rotary drills with 200 mm bit size for production drilling and down the hole (DTH) drills with 127 mm bit size for wall control drilling; diesel-powered RC drills for bench-scale grade control drilling; 15.5 m^3 bucket sized hydraulic excavators and 13 m^3 bucket sized wheel loaders for production loading; 140 tonne payload rigid-frame haul trucks and 40 tonne articulated trucks for production hauling; plus ancillary and service equipment to support the mining operations. In-pit dewatering systems will be established for the pit. All surface water and precipitation in the pits will be handled by diesel powered pumps.

Ore will be hauled to a crusher 0.5 km west of the pit and crushed to feed the process plant. Waste rock will be deposited into waste rock storage facilities (WRSF) 0.5 to 2.0 km west of the pit or used as rockfill to construct a tailings dam 4.0 km southwest of the pit.

Ultimate pit limits are split into phases or pushbacks to target higher economic margin material earlier in the mine life. The pits are split into nine distinct phases, with the initial phases containing mineralisation with a higher gold grade and lower strip ratio than later phases.

During the pre-stripping phase, all ore mined in the pit will be stockpiled. Cut-off grade optimisation on the mine production schedule will also send ore to a high-grade ore stockpile near the primary crusher. The stockpiled Mineral Reserves are planned to be re-handled and fed to the crusher once the pits are exhausted.

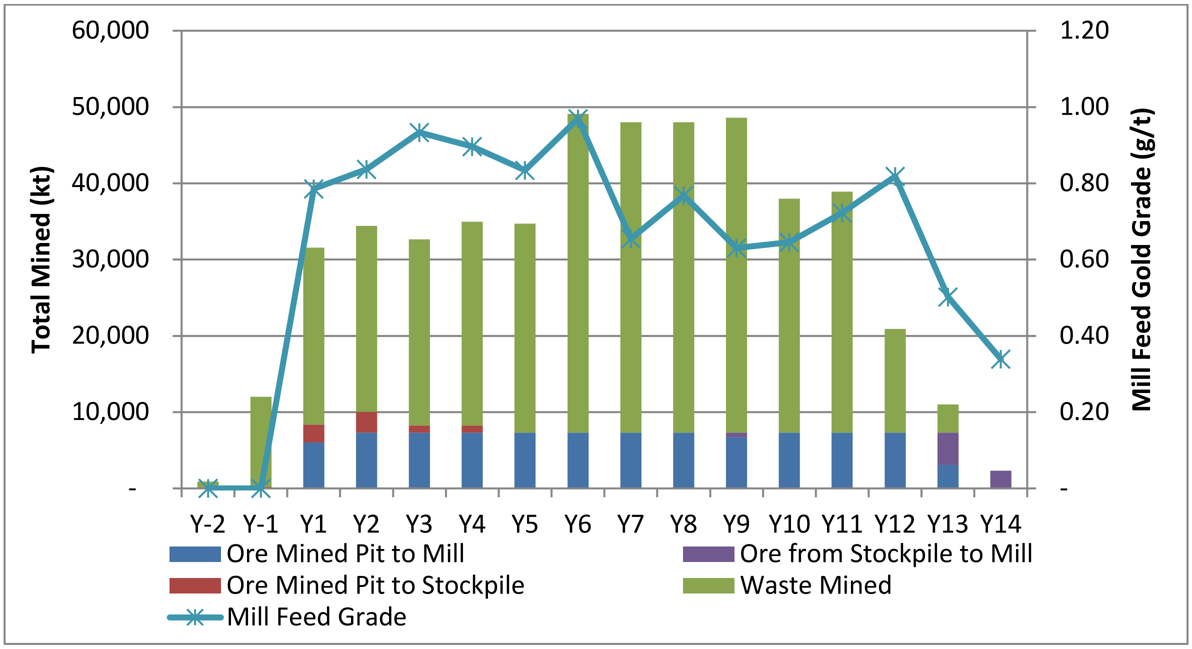
Mining operations will be based on 365 operating days per year with two 12-hour shifts per day. An allowance of 5 days of no mine production has been built into the mine schedule to allow for adverse weather conditions.

Maintenance on mine equipment will be performed in the field with major repairs to mobile equipment in the shops located near the plant facilities.

Annual mine operating costs per tonne mined range from $2.07 to $2.49/t with a LOM average of $2.22/t mined. Mine operations will include grade control and production drilling, blasting, loading, hauling, and pit, haul road and stockpile maintenance functions. Mobile equipment maintenance operations will also be managed by the Owner and are included in the mine planning and costs.

The mine equipment fleet is planned to be purchased via a lease financing arrangement.

Figure 1-2 summarizes the proposed ore and waste schedule for the 2021 PFS Mine Plan.



Source: Moose Mountain, 2021

Figure 1-2 Mine Production Schedule, Material Mined and Mill Feed Gold Grades

1.17 Recovery Methods

The process plant was designed using conventional processing unit operations. It will treat 20,000 tonnes/day or 913 tonnes/hour based on an availability of 8,059 hours per annum or 92%. The crusher plant section design is set at 70% availability. The plant will operate with two shifts per day, 365 days per year, and will produce 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 ball mill operating in closed circuit with a hydro-cyclone cluster. The SAG mill slurry discharges through a trommel where the pebbles are screened and recycled to a pebble crusher before returning to the SAG mill.

A primary gravity circuit treats 32% of the mill circulating load and is fed from the combined mill discharge stream. Gravity concentrate batches are treated in an intensive leach reactor, followed by electrowinning for recovery of gold and silver.

The hydro-cyclone overflow with P80 of 180 µm will flow to a four-stage flotation circuit including rougher flotation, rougher scavenger flotation, cleaner and re-cleaner flotation. Rougher scavenger flotation tailings will report to the NPAG tailings pond, cleaner and recleaner tailings are dewatered in a thickener and fed to a scavenger gravity circuit. Continuous gravity concentrators produce a concentrate at a mass pull of 5% which is added to the re-cleaner concentrate (overall mass pull of 3%) and fed to the concentrate regrind circuit. Combined flotation and scavenger gravity concentrates are ground to 80% passing 22 µm in a regrind mill before dewatering in a thickener to 50% solids ahead of the CIL circuit.

The concentrate leach-adsorption circuit consists of one pre-aeration tank and eight CIL tanks with a leach circuit residence time of 48 hours at 50% w/w solids. Oxygen is sparged into the pre-aeration and initial leach tanks to maintain adequate dissolved oxygen levels for leaching at 20 ppm. Hydrated lime is added to the pre-aeration tank and lime and sodium cyanide are added to the initial CIL tanks. Fresh/regenerated carbon from the carbon regeneration circuit is returned to the last tank of the CIL circuit and is advanced counter-currently to the slurry flow by pumping slurry and carbon. Slurry from the last CIL tank is pumped to the cyanide detox tank.

Gold and silver leached in the CIL circuit will be recovered onto activated carbon and eluted in a pressurized Zadra elution circuit and then recovered by electrowinning in the gold room. The gold–silver precipitate will be dried in a drying 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 CIL circuit.

CIL tailings will be treated for cyanide destruction. Gravity scavenger circuit tailings are combined with CIL tailings after cyanide destruction and pumped to the PAG tailings pond.

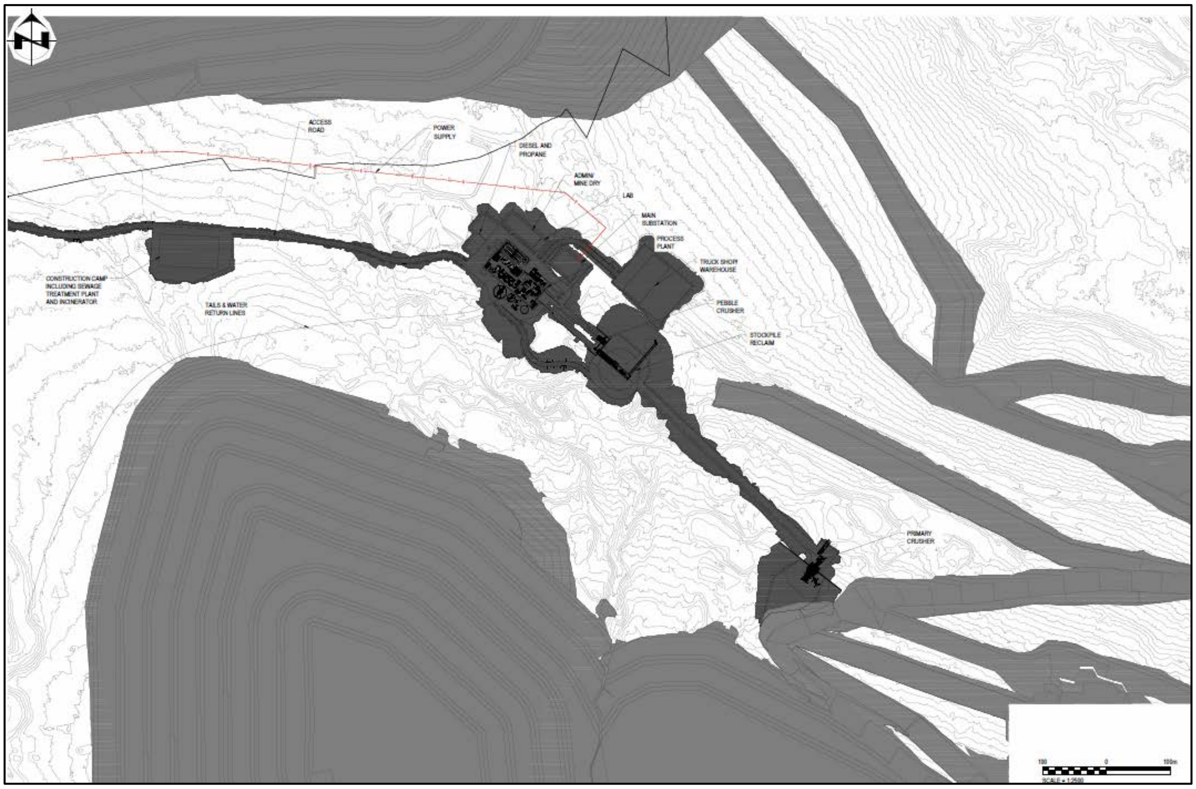
Raw water will be pumped from two freshwater wells to a raw-water storage tank. Potable water will be sourced from the raw water tank and treated in a potable water treatment plant. Gland water will be supplied from the raw-water tank. Process water primarily consists of reclaim water from the water management pond. Reagents will include hydrated lime, sodium cyanide, sodium hydroxide, copper sulfate pentahydrate, hydrochloric acid, sodium metabisulfite, activated carbon, flocculants, collector (PAX), frother (MIBC) and depressant (CMC) and liquid oxygen.

The installed power for the process plant will be 26.2MW and the power consumption is estimated to be 26.6 kWh/ton processed. The installed power for the G&A areas is 1.5MW and power consumption is 1.1kWh/t treated. Peak propane demand is estimated at 13.5 m^3 per day.

1.18 Project Infrastructure

1.18.1 SMG Access

Currently, SMG is accessed from Williams Lake via the Likely Road, which is a paved secondary road that leaves Highway 97 at 150 Mile House, approximately 16 km south of Williams Lake, and continues for 87 km to Likely. From Likely, the Property is accessed from the Spanish Mountain 1300 Forest Service Road (FSR). The access roads to SMG are presented in the following (Figure 1-3) Access Roads to SMG.



Source: Ausenco, 2021

Figure 1-3 Access Roads to SMG

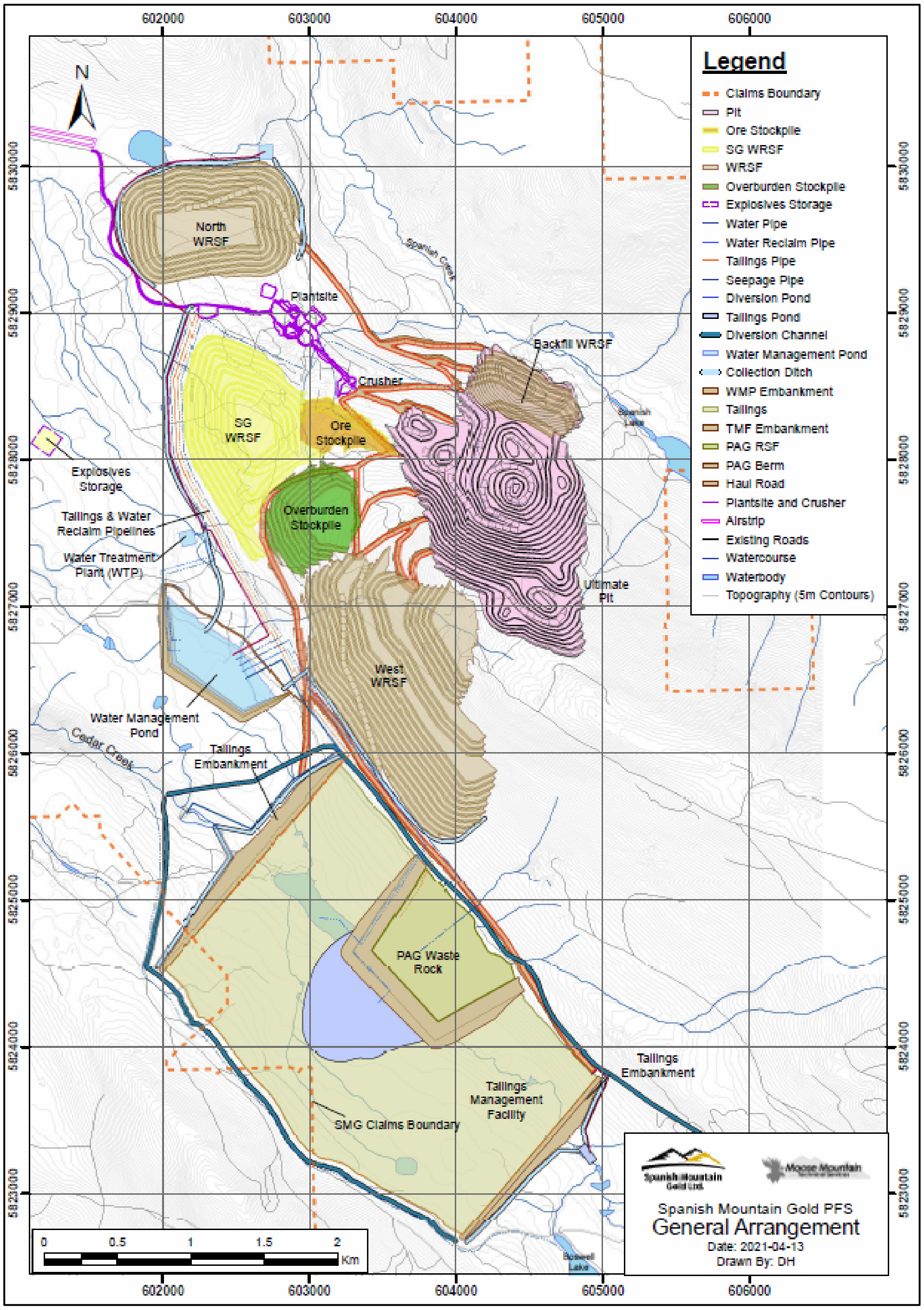
1.18.2 Mill Site Infrastructure

The process plant consists of ore stockpiling, crushing, conveying, grinding, gravity and intensive leaching, flotation, flotation concentrate regrinding and thickening, CIL leaching and adsorption, acid washing of loaded carbon, elution cyanide detoxification, carbon regeneration, thickening and cleaner tailings gravity concentration, and reagents. Refer to Section 17 for details of the process facilities. The following is a general description of the ancillary facilities at the process plant site. The infrastructure planned at the process plant to support the mining and processing operations includes:

- Upgraded and new access roads, and new site roads
- Administration facilities, mine dry, truck shop, and maintenance facilities
- Assay laboratory/cold storage building
- Waste treatment systems
- Solid waste disposal facilities
- Tailings storage facility
- Overall site water management
- Water treatment facilities

- Temporary construction camp
- Power supply and distribution system
- Site services
- Fuel
- Propane
- First aid station
- Water supply
- Communication system.

Figure 1-4 shows the overall general layout of the SMG.



Source: MMTS (2021)

Figure 1-4 General Arrangement – Main SMG Facilities

1.18.3 Tailings Storage Facility and Surface Water Management

The principal objective of the Tailings Storage Facility (TSF) is to provide secure containment of all tailings solids and potentially acid generating (PAG)/metal leaching (ML) waste rock. The metallurgical process involves a gravity circuit followed by a rougher flotation circuit to produce rougher tailings. The process feed is reground and subjected to carbon-in-leach and cyanide detoxification circuits before being combined with a pre-float concentrate to produce the cleaner tailings and detox tailings stream, which is assumed to be PAG and ML if allowed to oxidize. The tailings streams will be transported from the plant site to the TSF in separate pipelines. The rougher tailings will be discharged from spigots located at the embankments and the west side of the TSF forming extensive drained tailings beaches. The cleaner tailings and detox tailings stream will be pumped and discharged sub-aqueously into a separate location within the facility, referred to as the PAG Cell.

The TSF will comprise a north embankment and a south embankment. The embankments will be zoned earthfill/rockfill structures with a low-permeability core for seepage management. The embankments will include filter and transition zones to ensure proper filter relationships between adjacent zones, and to convey drainage within the embankment. A downstream shell zone, which comprises most of the embankment material, will be constructed with Non-PAG mine waste. The TSF embankments will be expanded in stages throughout the mine life using the centerline construction method, with each stage providing the required capacity for the period until the next stage of construction is completed.

The ultimate TSF was sized to store approximately 92 Mt of tailings, 66 Mt of PAG waste rock, the Inflow Design Flood, an operational supernatant pond, plus freeboard. The TSF starter embankment which will be constructed during the pre-production phase, has been sized to store the estimated volume of tailings and PAG/ML waste rock produced during the first two years of operation.

The project includes a separate WMP which will serve as the primary site water management component during operations. Site runoff, including the water in the TSF supernatant pond, is pumped to the WMP. This significantly reduces the volume of water stored within the TSF. The WMP provides a buffer for the water treatment plant to reduce the peak flows requiring treatment.

Tailings from the Process Plant will be delivered to the TSF in two different streams, a rougher tailings stream and a cleaner and detox tailings stream. The rougher tailings distribution system conveys tailings from the Process Plant for discharge into the TSF from the North and South Embankments, and the west side of the TSF, throughout operations. The rougher tailings will be discharged into the TSF from a series of large diameter valved off-takes located along the embankments. The cleaner tailings will be discharged separately to allow subaqueous deposition within an internal PAG Cell.

1.18.4 Water Management

Site water management involves controlling surface water around the SMG site during the construction, operations, closure, and post-closure phases of the SMG project.

1.18.5 Water Treatment Plant

Treatment of mining-influenced waters (MIW) will be required during all stages of the Project. During the initial Pre-Production year Y-2), SMG will manage the suspended solids generated during earthwork and construction through best practices and water management; active treatment is not required in the first year. In the second Pre-Production Year, Y-1, active treatment will be required to treat nitrogen compounds derived from blasting used to generate clean rock for construction; SMG will continue to manage suspended solids without active treatment. Beginning Production Year 1 and continuing through active closure (Y15 through Y18, active treatment of MIW will be required. In the active closure period, active treatment will transition to the passive treatment of MIW.

SMG will construct a Water Treatment Plant (WTP) to actively treat influent from the WMP, which will receive and store MIW generated from the active portions of the Site. The WTP will incorporate a variety of process technologies including oxidation, settling and clarification, microfiltration (MF), denitrification, and reverse osmosis (RO). The WTP effluent will discharge to Cedar Creek. Design and construction of a 15,000 m^3/day WTP will occur in Y-2, with operations beginning in Y-1. The WTP will initially be equipped to treat nitrogen compounds (blasting residuals) in Y-1. In Y1, the WTP will be fitted with additional treatment equipment to address MIW generated during mining including removal of parameters such as sulphate and metals from the WMP influent. The WTP capacity will be expanded to 18,000 m^3/day in Y6. The WTP will continue to operate through the active closure period (AC1 through AC4). Active treatment will cease in AC4 and the WTP will be dismantled and transported offsite in late AC4.

Separate passive treatment systems (PTSs) will be constructed to address seepage from the following specific sources: The North and West Rock Storage Facilities (RSFs), the North Seepage Collection Pond (SCP), and the Pit Lake. The PTSs will include combinations of iron terraces, biochemical reactors (BCRs), and aerobic polishing wetlands. Effluent from the West RSF and North SCP PTSs will be directed to Cedar Creek, while that from the North RSF will be directed to Spanish Creek. Primary treatment of the Pit Lake will occur in-situ, with a downstream iron terrace treatment unit for polishing before discharging treated Pit Lake water to Spanish Creek. Design of the North RSF, West RSF, and North SCP PTSs will occur in Y4 and Y5, with construction occurring in Y6 and commissioning beginning inY7. Design of the Pit Lake PTS will occur inY11 andY12, with construction occurring in Y13. The timing of the Pit Lake PTS construction is considered conservative because the Pit Lake will not fill its spill point into Spanish Creek until Y25.

1.18.6 Power and Electrical

The mill throughput is nominally 20, 000 t/d. At this production level, the plant load is estimated to be approximately 40 MW ± 10%. BC Hydro will establish a new 138 kV Substation near Highway 97, to be known as SMMX Substation. The new substation will be fed at 230 kV from existing BC Hydro line 2L95. The new substation will contain a 30 MVA step-down transformer to 138 kV, and metering at 138 kV. A set of outgoing suspension insulators will form the Point of Interconnection (POI). The new substation will be owned and operated by BC Hydro. Everything downstream of the POI will be owned and operated by Spanish Mountain.

A new 138 kV transmission line will be installed between the POI and the receiving substation at the Spanish Mountain mine site. The transmission line will be single pole type. The transmission line will generally follow the road, to a point just west of the town of Likely, then it will be routed north, around Likely, to the mine site. The transmission line will be owned and operated by Spanish Mountain.

A new 138kV Receiving Substation including 138kV disconnect switch, circuit breaker, transformer and capacitor bank will be constructed. On-Site distribution will be installed at 13.8 kV.

1.19 Market Studies and Contracts

The Project will yield gold doré as its final product, which is expected to be sold on the spot market through marketing experts retained by SMG.

A gold price of US$1,600 per ounce, and a silver price of US$24 per ounce, have been used for the 2021 Spanish Mountain Gold Prefeasibility Study. Spanish Mountain Gold plans to contract out the transportation, security, insurance, and refining of doré. Smelter terms / offsite costs related to the sale of doré and used for the study are shown in Table 22-1. An exchange rate of C$1.00: US$0.76 was used, based on the trailing three years C$: US$ foreign exchange rate.

1.20 Environmental Studies, Permitting, and Social Community Impact

Environmental baseline studies were conducted between 2007 and 2012 in support of the environmental assessment (EA) for the project that was previously proposed and were re-started in 2020 to support entry to a new EA process.

Baseline environmental data collection is well advanced, including:

- Meteorology
- Air quality
- Surface water quality
- Surface hydrology
- Ground water quality
- Hydrogeology
- Fish and fish habitat
- Vegetation and wildlife
- Heritage resources

Discussions with government regulatory agencies were undertaken to develop methods to avoid or mitigate negative environmental effects. None of the environmental parameters identified to-date are expected to have a material impact on the ability to extract the mineral resources or reserves.

SMG will be required to re-start the environmental assessment process under the new provincial and federal legislation enacted in 2019.

Upon completion of the provincial and federal EAs, SMG will secure the required permits and authorizations. Major mines require several authorizations from many different provincial and federal agencies. The key Project permits are expected to be:

- Provincial Mines Act permit
- Provincial waste discharge authorizations under the Environmental Management Act
- Federal Fisheries Act Authorizations

- Amendment to Schedule 2 of the Federal Metal and Diamond Mining Effluent Regulations (MDMER).

Several other permits will also be required, including licences or approvals for water use or water storage facilities under the Water Sustainability Act and the B.C. Dam Safety Regulation, authorizations for any cutting or spoiling of crown trees under the Forest Act, and land tenure under the Land Act.

The environmental and community aspects of the Project have been well studied and are understood. Ongoing and updated studies continue to add clarity to the baseline condition and will be used as a foundation to re-enter the environmental assessment and permitting processes. There are no known environmental or social constraints that could materially impact the issuer's ability to extract the mineral resources or mineral reserves.

1.21 Capital and Operating Costs

The capital cost and operating estimates for the SMG are developed to a level appropriate for a Prefeasibility Study. All capital and operating costs are reported in Canadian dollars (C$) unless specified otherwise. The overall capital cost estimate (except for the Water Treatment estimates – which is a Class 5 AACE estimate) meets the American Association of Cost Engineers (AACE) Class 4 requirement of an accuracy range between −25% and +25% of the final Project cost.

SMG benefits from significant existing infrastructure, which helps reduce the initial capital cost. Total initial pre-production capital cost is $607.2M inclusive of construction indirect costs, engineering-procurement-construction-management (EPCM), contingencies and owners' costs. The sustaining capital is $290.5M inclusive of mine development capital, and process plant. The LOM capital expenditure (CAPEX) is $897.7M exclusive of closure costs. Total Closure costs is $159.6 including reclamation of the open pit, process, and non-process facilities, tailing storage facility, surface site water management, environmental items, project indirects Owners costs, and contingencies. Open Pit mining and haulage are anticipated to be completed using an owner-operator development model operating 365 d/a with a mobile equipment fleet which is included in operating costs. Table 1-3 presents the Project capital cost breakdown and the costs for each work breakdown structure (WBS).

Table 1-3 Total Project Capital Cost Summary by Area

WBS	Description	Total Cost ($ '000s)		
		Initial	Sustaining	LOM Total
1 Direct Costs				
10	Overall Site Development	25,600	1,950	27,550
20	Mining	73,438	163,731	237,169
30	Ore Handling	33,743	-	33,743
40	Process	125,527	8,057	133,584
50	Tailings and Water Management	40,318	35,611	75,929
60	Environmental	1,600	7,200	8,800
70	On-Site Infrastructure	41,617	-	41,617
80	Off-Site Infrastructure	64,170	3,250	67,420
82	Water Treatment Facilities	10,347	35,006	45,353
2 Indirect Costs				
90	Project Indirect Costs	101,964	18,510	120,474
3 Owner's Costs				
98	Owner's Costs	13,654	-	13,654
4 Contingency				
99	Contingency	75,225	17,175	92,400
Total Project Costs		**607,203**	**290,490**	**897,693**

The total estimated initial capital cost is $607.2M (and sustaining capital of $290.5M) is $897.7M over the LOM. The estimated LOM operating costs are $19.38/t of mill feed.

1.21.1 Operating Costs

LOM operating costs for the Project were developed from first principles for mining, processing, site services, and administration using the mine and processing plans, incorporating development rates, labour, materials, consumables, and certain contract services for a 20,000 t/d processing rate. Table 1-4 shows the breakdown of LOM operating costs.

Table 1-4 Project LOM Operating Costs

Operating Costs	Costs ($/t milled)
Mining	10.80
Processing and G&A	7.67
Water Treatment	0.47
Tailings and Water Management	0.17
Owners G&A	0.27
Total Operating Costs ($/t milled)	**19.38**

1.21.2 Closure Costs

Mill, TSF, water management, water treatment and infrastructure closure estimates have been prepared as of the date of this report. The closure cost for the SMG is $159.6M based on estimates summarized in Table 1-5.

Table 1-5 Closure Cost

Description	Closure Cost ($ '000s)
Direct Closure Costs	
Mining	32,127
Process and on-site infrastructure	14,690
TSF and Water Management	62,222
Water Treatment	24,958
Environmental (including passive closure)	2,600
Directs Subtotal	**136,597**
Project Closure In-direct Costs	
Owner's Costs	4,790
Indirect Costs	2,593
Contingency	
Contingency	15,622
Total Closure Costs	**159,602**

1.22 Economic Analysis

An economic evaluation of the Project is carried out incorporating all the relevant capital, operating, off-site, working, and sustaining costs, and royalties.

For the 14-year project life, and 95.9 Mt Reserve inventory, the following pre-tax financial parameters are calculated:

- 25% IRR
- 3.2-year payback on $607M capital
- $848M NPV at 5% discount value.

The following post-tax financial parameters were calculated:

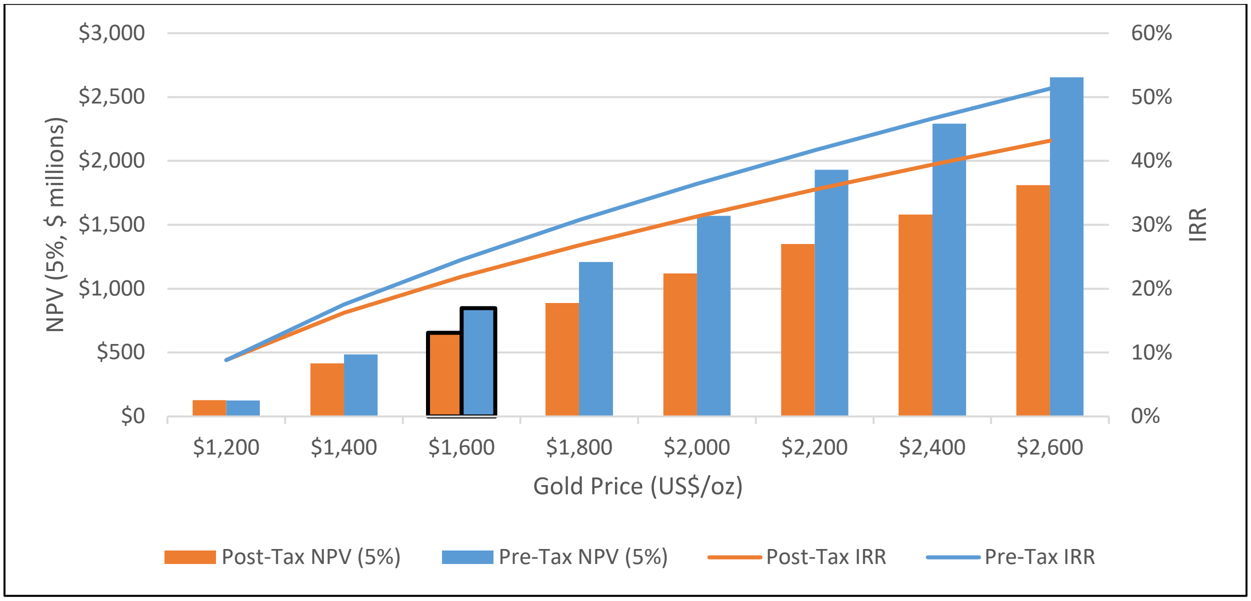
- 22% IRR
- 3.3-year payback on $607M capital
- $655M NPV at 5% discount rate.

The following parameters are used for the financial analysis:

- Gold price of US$1,600/oz.
- Silver price of US$24/oz.
- Exchange rate of US$0.76 to C$1.00.
- 99.8% payable gold and 90% payable silver.
- US$1.00/oz. gold refining charges, and US$0.60/oz. silver refining charges.
- US$1.00/oz. transport charges on produced gold and silver

- 0.15% insurance on value of produced gold and silver
- 1.5% NSR royalty.

A sensitivity graph based on various gold prices is set out in Figure 1-5 below:



Source: Moose Mountain, 2021

Figure 1-5 Project Economic Sensitivity

1.23 Adjacent Properties

There are no active exploration properties immediately adjacent to the Spanish Mountain Property. The Property is in an area that has seen active past exploration and mining activity for alkaline porphyry copper-gold deposits that are completely segregated from the Spanish Mountain Gold Property and are not in any way indicative of the mineralization on the Property.

Currently, the most advanced property in the area is Imperial Metals' Mount Polley Mine, which is alkalic porphyry copper-gold deposit located about 15 km to the west. As of December 31, 2018, the deposit had measured and indicated resources of 194M tonnes grading 0.29% copper and 0.29 g/t gold (Imperial Metals website).

The QR Mine is a propylitic gold skarn located 24 km northwest of the Property. As of July 2009, the West Zone had a measured resource of 40,000 tonnes grading 3.65 g/t Au and an Indicated resource of 479,000 tonnes grading 4.18 g/t Au, all at a cut-off grade of 2.0 g/t Au (Fier et al., 2009).

1.24 Other Relevant Data and Information

The current plan will be to complete interim studies and fieldwork before executing a Feasibility Study by Q4 2023 and to proceed to detailed engineering. This will allow SMG to be able to start early works at site in Q1 2025 (Y -2). This work will include early construction of the earthworks for the water treatment facilities and the installation of 138 kV transmission line from Highway 97 to the new Main Substation.

Upon Project approval starting in Q1 of Y-2, the balance of the facilities will be constructed. The total Project is expected to require 24 months, resulting in Project start-up in Q1 of Year 1 (2027).

1.24.1 Planning and Scheduling

The project execution schedule has been developed for the SMG project to achieve production by Q1 2027. The schedule was developed with interaction between permitting requirements, the mine development team, the processing and surface infrastructure team and the checking for timing of the production ore and interactions with the surface infrastructure and tailings facilities upgrade scopes of work.

The schedule considered the early ordering of equipment using the designs developed for the Feasibility study in which key long lead items were tendered. These long lead items were reviewed to the point of a technical recommendation for SMG.

Construction activities are dependent on receiving an approved *Mines Act* EMA (MEMA) Permit, Fisheries Act Authorization and a Metal and Diamond Mining Effluent Regulation (MDMER) permit.

The major Project activity durations and milestones are listed below in Table 1-6.

Table 1-6 Major Milestones

Milestones (Completion Dates)	Milestone Start Date	Milestone Completion Date
BCH SIS Review and Approval	Q2 2020	Q4 2020
Detailed Engineering	Q1 2025	Q4 2025
Long Lead Orders – Award	Q2 2025	Q3 2025
Construction Permits Approved	Q1 2025	Q1 2025
Construction of Plant Site and Infrastructure	Q2 2025	Q4 2026
Plant Commissioning	Q3 2026	Q4 2026
Construct Spanish Lake Access Road	Q1 2025	Q3 2025
Commence Mining Development and Mining	Q1 2025	Q4 2026

Activities also allow for the construction of the Spanish Lake Road which will commence in Year -2 to facilitate road access around the mine by the public.

The current plan will be to complete this Prefeasibility Study by Q2 2021 and to proceed to trade-off and other environmental and engineering studies prior to commencing the Feasibility Study. Early Works will include preparing the mill and ancillary facilities, early construction of the earthworks for the new WTP, and installation of the new 138 kV transmission powerline from Highway 97 to the new Receiving Substation on site.

1.25 Conclusions and Interpretations

This Prefeasibility Study represents an economically viable, technically credible, and environmentally sound mine development plan for the SMG project. The project benefits significantly from its proximity to existing sources of labour and existing infrastructure. The Prefeasibility Study fourteen-year LOM plan demonstrates positive economics. Industry-standard mining and processing methods were used in the study and the QPs are not aware of any fatal flaws that encumber the SMG project from undergoing further economic studies, permitting, financing, and ultimately development. It is recommended that the Project proceeds to permitting and feasibility level engineering design.

The most significant potential risks associated with the Project are uncontrolled mining dilution, operating and capital cost escalation, permit acquisition, reduced metallurgical recoveries, unforeseen schedule delays, changes in regulatory requirements, the ability to raise financing, exchange rate, and metal prices. These risks are common to most mining projects, many of which may be mitigated, at least to some degree, with adequate engineering, planning, and proactive management.

The initial capital cost of the Project is estimated to be $607.2M Initial Capital, $290.5M in Sustaining Capital. The Closure Costs for the Project is estimated to be $129.6M. The LOM AISC total cost is US$801/oz. The project NPV (after-tax) is estimated to be $655M using a 5% discount rate. The IRR (after-tax) is estimated to be 22% with a payback period (after-tax) of 3.3 years.

Mineral Reserves

Factors that may affect the Mineral Reserve estimates include metal prices, changes in interpretations of mineralisation geometry and continuity of mineralisation zones, geotechnical and hydrogeological assumptions, ability of the mining operation to meet the annual production rate, operating cost assumptions, process plant and mining recoveries, the ability to meet and maintain permitting and environmental licence conditions, and the ability to maintain the social licence to operate.

Mining

Reasonable open pit mine plans, mine production schedules, and mine capital and operating costs have been developed for the Mineral Reserves estimates at Spanish Mountain Gold.

Pit layouts and mine operations are typical of other mountainous open pit gold operations in Canada, and the unit operations within the developed mine operating plan are proven to be effective for these other operations.

The mine plan supports the cash flow model and financials developed for the Prefeasibility Study.

Process and Infrastructure

The Spanish Mountain Gold completed numerous comminution and metallurgical testwork programs to support the PFS. This included head grade analyses, a full suite of comminution tests, flotation, gravity separation, and leach tests and cyanide detoxification.

Tests were performed on mineralization that is representative of the 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 over the life of mine. The grade variability samples gold and silver grades ranged from 0.15–3.25 g/t Au and 0.5–4.4 g/t Ag.

Selection of primary grind size should be revisited in future studies as the P80 size of 180 µm was selected at a time when the lower gold price did not justify finer grinding.

Tailing Storage Facility and Water Management

- The TSF has the capacity to store the tailings and PAG waste rock produced for the 14-year mine life.
- There is sufficient material available to construct the TSF embankment with most of the construction material being sourced from the pit stripping activities.
- The site investigation work completed at the TSF is sufficient for a PFS study.
- The two tailing streams produced in the mill are effectively managed at the TSF. The rougher tailings will be discharged from spigots located at the embankments and the west side of the TSF forming extensive drained tailings beaches. The cleaner tailings and detox tailings stream will be pumped and discharged sub-aqueously into the PAG Cell. a separate location within the facility, referred to as the PAG Cell.
- Managing the PAG tailings and waste rock in a separate internal cell facilitates the development of large tailings beaches in the TSF and the subaqueous management of the PAG materials in the PAG Cell.
- Site runoff is stored in the Water Management Pond which has been sized to store the site runoff from the 95th percentile wet year assuming a water treatment rate corresponding to approximately the 80th percentile of annual site runoff.
- The site plan includes diversions where required to minimize the amount of non-contact water entering the project site.
- The TSF design allows for effective reclamation of the tailings surface with minimal surface ponding post closure.

Off-Site Infrastructure

No closure costs were applied to the 138kV electrical transmission line and the on-site receiving substation as they could be of benefit to the local community.

Geochemistry

Geochemical characterization for the PFS was based on the 2012 SRK study that indicated:

- The proportion of rock with ARD potential at Spanish Mountain is relatively low.
- The potential for metal leaching at neutral pH however was identified regardless of ARD potential.
- The rougher tailings generated from metallurgical testing had low ARD and metal leaching potential, and the cyanide leach tailings indicated a high potential for ARD and metal leaching.

- Results indicate that management measures will be required to reduce the generation and effects of ARD/ML.

Water Treatment

Water treatment will be required during all years of the Project, with the exception of the first Pre-Production Year (Y-2) when best practices and water management will be employed to address suspended solids generated by construction. Beginning in Y-1, active treatment will be employed to remove deleterious parameters from site waters. Active treatment will continue through Y18, transitioning to passive treatment during the Active Closure period. The treatment systems are conservatively designed to meet applicable water quality standards at the end-of-pipe; no assimilative capacity in the receiving streams (Cedar and Spanish Creeks) is assumed.

Separate PTSs will be constructed to address seepage from the following specific sources: The North and West Rock Storage Facilities (RSFs), the North Seepage Collection Pond (SCP), and the Pit Lake as described in Section 1.18.

1.26 Recommendations

The SMG project is well suited for a potential mining operation. A Prefeasibility Study–level fourteen-year LOM plan has a positive economics, and it is recommended that the Project proceed to permitting and detailed design.

A summary of the suggested work program to support the next phase of the Project includes the following components:

Mineral Resource Estimate

During the drillhole database validation exercise, a survey of drillhole collars carried out in 2008 by Allnorth indicated different collar coordinates for 48 holes. From further investigation it was found that these holes were re-surveyed after this initial survey and that the collar coordinates in the drillhole database are the correct ones. Survey checks for a few drillholes carried out in December 2020 have confirmed the current coordinates. It is recommended that all the 48-hole collars be re-surveyed during the 2021 summer months and documented to further confirm their collar coordinates.

Marc Jutras recommended that the broader geologic units could be refined for future updates.

A drilling campaign was carried out in November and December of 2020 and was not incorporated into this mineral resource estimate due to delays in the assaying process. It is recommended that an update of the mineral resource estimate be carried out with these additional drillholes.

The budget for the above recommendations by Marc Jutras is approximately $62,000, and is detailed in the following Table 1-7:

Table 1-7 Recommendation Work Details

Task	Description	Cost
Re-Survey of Drill Holes	Confirmation survey of 48 drill hole collars from the Allnorth surveying campaign of July 2008	$4,500
Update of the Mineral Resource Estimate with the Nov/Dec 2020 Drillholes	Update of the geology model with new drill holes and refining the interpretation of the geologic units.	$19,500
	EDA update: compositing, capping, statistical plots	$4,000
	Redo variographic analysis	$8,000
	Au and Ag grade interpolation	$6,000
	Validation of Au and Ag grade estimates	$6,000
	Tabulation of mineral resources	$4,000
	Writing of report	$10,000
	Total	**$62,000**

Mining

MMTS recommends that following testwork and analysis to advance the project to Feasibility level engineering designs:

- The following geotechnical and hydrogeological testwork and analysis to bring the pit slope designs to a Feasibility level design:
    - Geotechnical logging of core holes targeting shallow faults, which are currently limiting the PFS design.
    - Large-scale geotechnical outcrop mapping to characterize the large-scale roughness, spacing and persistence of the controlling fault sets.
    - Delineate watershed areas directing surface water into the proposed pit and investigate the drainage features evident near the east and west walls.
    - Investigation and numerical simulation of the degree of hydraulic connection between the north end of the open pit and Spanish Creek. Conduct ground geophysics at the base of the Spanish Creek valley, to better characterize the depths of alluvial sediments and fractured bedrock.
    - Investigation and numerical simulation of the depressurization requirements of the updated open pit.
    - A detailed geological model, including folding, faulting, and a reconciliation of the various interpretations on the deposition and tectonic history of the deposit.
- Further waste rock geochemical characterization, with targeted updates to source terms and PAG definition.

- Geotechnical analysis of the foundations identified for the WRSF's.
- Condemnation drilling of the footprints identified for the WRSF's, and site infrastructure should be carried out.
- Execute a grade control drilling and interpretation program, or possibly even a bulk sample program, in selected areas of the on deposit that are planned to be mined for initial mill feed. The resultant tonnes and grade from this interpretation should be compared to the equivalent area resource modelled tonnes and grade. Results to be incorporated in ongoing grade control strategy and mine planning.
- Drill and blast testing to be carried out by drilling vendors and local explosives suppliers by analysing local rock types and conditions to assess the achievable drill penetration rates, optimal explosives mix and target powder factor for use in this operation.
- Updating of all mine planning work to incorporate results from other recommended studies, including optimization studies for pit limits and mine scheduling, and various operational trade-off studies (contractor vs. owner fleet, lease vs. purchase, etc.).
- The approximate costs of the above recommendations are as follows:
    - Geotechnical drilling and hydrogeological pumping tests = $2.8M
    - Geotechnical and hydrogeological field work, lab work and engineering for FS level design = $1.4M
    - Condemnation drilling = $1.0M
    - Grade control drilling or bulk sampling = $3.0M
    - Remaining engineering and analysis = $0.3M

Process and Infrastructure

The Metallurgical testwork recommended in support of feasibility study is as follows:

1. Primary grind size assessment
2. DFR rougher pilot study on gravity tailings for confirmation of predicted performance (recovery, mass pull and TOC rejection, and mineral analysis by size of rougher concentrate)
3. Bench scale cleaner tests on DFR rougher concentrate to evaluate:
    - second cleaner stage included or excluded
    - open vs closed circuit for cleaners
    - gold by size analysis of cleaner tailings
    - gravity scavenger test performance
4. Bulk concentrate generation to support equipment sizing and reagent consumption, specifically:
    - Determination of vendor specific energy for regrind
    - Dynamic settling tests – cleaner tailings and concentrate after regrind
    - Slurry rheology - concentrate slurry at P80 ± 20 microns and 35-50% solids
    - Leach time evaluation
    - Oxygen uptake test

5. Variability sampling to provide spatial coverage of the ore body, testing at optimised flowsheet conditions to include the following:

- Chemical characterisation
- Determination of comminution properties (SMC and BBWI, RWi, Ai data)
- Gravity/Flotation/scavenger gravity/regrind/CIL performance
- Grade vs recovery relationships for varying S, TOC, and Au grades

6. Tailing characterisation – flotation and leach tailings (determination of chemical and mineral composition of solids, chemical composition and water quality parameters for solutions, and cyanide speciation of detox solution)

- Five engineering trade-off studies are recommended as follows:
- Primary grind size evaluation
- Regrind size evaluation and technology selection – to inform test work program
- Primary gravity circuit evaluation – to inform test work program
- Scavenger gravity circuit evaluation – to inform teste work program
- Trade-off of flotation recovery vs mass pull and corresponding concentrate processing circuit capital and operating costs

The cost of testwork (excluding drilling) is expected to range from $ 400,000 - $450,000, dependent on the final variability sample selection. Testwork management costs are estimated at $75,000, and engineering trade-off studies at $50,000.

The cost for a feasibility study covering the process and infrastructure scope is approximately $1.8M.

Estimated cost of a Feasibility Study for the process and infrastructure is estimated at $1.5M.

On-site Infrastructure

- The QP recommends a site-wide civil geotechnical investigation to verify ground bearing pressures, soil parameters, ground dynamic modulus of elasticity, and settlement parameters: the approximate cost of this work is $250,000.
- A route survey report is recommended for the next phase, prior to construction, at an approximate cost of $30,000.

Off-Site Infrastructure

- Confirm viability of installing 138 kV pole line alongside of road between BC Hydro substation and site. (Budget estimate $20,000)
- Investigate possibility of installing 138 kV pole line through Likely (thus saving 10 km of transmission line). (Budget estimate $20,000)
- Issue detailed line design to electrical contractor.
- Perform a LIDAR survey of the 138kV transmission line route. (Budget estimate $20,000)
- BCH ongoing engineering co-ordination related to the System Impact Study (Budget estimate $5000)
- Off-Site electrical Feasibility Study estimated budget $70,000

- Investigate local community benefits of the continuous service of the 138kV overhead distribution line and receiving site substation. Estimated budget for the investigation $15,000

Tailing and Water Management

- Complete additional site investigation programs to support the level of detailed required for future studies (Estimated Budget - $3.0M).
- Completing a site-specific Seismic Hazard Analysis (SHA) to confirm the seismic design parameters. (Estimated Budget - $0.1M).
- Develop geochemical source terms and a site water quality model, (Estimated budget - $0.10M).
- Completing a surficial landform mapping desktop study. This requires higher resolution LiDAR data that what is currently available for the site. (Estimated Budget $0.10M).

Geochemistry

Recommended geochemistry related tasks for post-PFS studies to support feasibility and permitting for the Spanish Mountain Gold Project are as follows:

- Expanded static and humidity cell test characterization program on waste rock from the eight geological domains within the proposed open pit (Currently in progress).
- Addition of a saturated column to the waste rock characterization program to assess the loading from submerged potentially acid generating (PAG) waste rock.
- Geochemical characterization of ore feed, tailings and process water generated from the upcoming pilot plant metallurgical testing program.
- Static test characterization of overburden and borrow sources that may be used in construction/infrastructure for the project.
- Development of source terms that will be used as inputs into the site-wide water and load balance (undertaken by others) to generate water quality predictions to support water treatment evaluations and permitting.

The estimated budget for these recommendations is $370,000.

Environmental Studies, Permitting, and Social or Community Impact

- KP recommends that SMG reinitiate baseline studies and re-enter the provincial and federal environmental assessment processes (Estimated budget $3.0M).
- Secure permits and authorizations from government and regulatory agencies.

Water Treatment Process

The water treatment processes planned for the Project are conventional, proven technologies that are commonly implemented to treat mine and tailings water produced at open pit gold mine operations. The projected volumes of water to be treated, both by active and PTSs, are very large. Additional studies and design advancements (the use of dewatering wells in advance of pit development, construction of less permeable caps on rock storage facilities and/or use of impermeable interlayers to decrease infiltration, and progressive reclamation and optimized water management to decrease the volume of surface water requiring treatment) to reduce the generation of MIW could decrease the volume of impacted water to be treated, which would lower both capex and opex. In addition, source control and/or selective waste rock handling to reduce the concentrations of parameters requiring treatment could lower both capex and opex. Recommendation to further assess the demolition, haulage, disposal/salvage of the WTP should be performed to refine the closure costs associated with the structure and associated water treatment equipment.

The costs of investigations (e.g., bench testing, kinetic testing) to evaluate the effectiveness of bactericides at inhibiting pyrite oxidation is estimated to be approximately $125,000.

2 INTRODUCTION

The purpose of this Technical Report is to present the results of the PFS of Spanish Mountain Gold's mineral resource property located in British Columbia, Canada. Spanish Mountain Gold Limited (SMG, the Owner) is a Canadian-based exploration and development company based in Vancouver, Canada. Shares of SMG are currently traded on the TSX. SMG is focused on starting the Spanish Mountain Gold Project (SMG).

The Project involves the development of a gold deposit located in south-central BC, Canada, approximately 6 km southeast of the community of Likely and 66 km northeast of the City of Williams Lake.

The Project is situated between Quesnel Lake and Spanish Lake; its centre is located at approximately latitude 52˚ 34' north and longitude 121˚ 28' west. The gold concentrator for the Project has been designed to process a nominal 7,300,000 t/a (or 20,000 t/d) of gold and silver bearing material from an open pit operation and will produce gold-silver doré as a final product.

All currency amounts are referred to in Canadian dollars ($ or C$) unless stated otherwise.

All units in this Report are SI (International System of Units) and Universal Transverse Mercator (UTM). Coordinates in this report and accompanying illustrations are referenced to North American Datum (NAD) 1983, Zone 10.

2.1 Terms of Reference

To complete the PFS, SMG engaged a team of independent consultants. The Technical Report was co-ordinated by Moose Mountain Technical Services (MMTS) in conjunction with Discovery Consultants (Discovery) for geology; Ginto Consulting Inc., (Ginto) for resources; pHase Geochemistry (pHase) for geochemistry; Ausenco Engineering Canada Inc., (Ausenco), for metallurgy, processing, on-site infrastructure, on-site power distribution, off-site infrastructure; and Knight Piésold (KP) for the Tailing Storage Facility, Overall Site Water Management and Environmental, MCA for the on-site main substation, main transmission powerline and partial on-site electrical distribution; Linkan Engineering for the Water Treatment Plant and PTSs; and is written to comply with standards set out in National Instrument (NI) 43-101 for the Canadian Securities Administration (CSA).

The Technical Report is a technical summary of available geologic, geophysical, geochemical, metallurgical, and diamond and reverse circulation drillhole information; as well as PFS level engineering and costing on the open pit, process facilities, site infrastructure, off-site infrastructure, tailings facilities and water management facilities. The authors, in writing this report, used sources of information as listed in the references listed in Section 27.0.

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

- Bill Gilmour, P.Geo., Geology, Co-Founder of Discovery
- Marc Jutras, P.Eng., Resource, Principal, Mineral Resources, Ginto
- Marc Schulte, P.Eng., Mining, MMTS
- Frank Grills, P.Eng., Principal Project Manager, MMTS
- Malcolm Cameron, P.Eng., Electrical Engineer, MCA
- Paul Staples, P.Eng., VP & Global Practice Lead, Comminution, Ausenco
- Les Galbraith, P. Eng., Specialist, Associate, KP
- Sam Billin, P. Eng., President/Founder, Linkan Engineering
- Andrea Samuels, P.Geo., Senior Geochemist, pHase Geochemistry

2.2 Qualified Persons

The following companies contributed to this technical report and provided Qualified Person (QP) sign-off for their respective sections (Table 2-1). All supporting documents cited in this report are referenced in Section 27.

Table 2-1 Prefeasibility Study Sections and Parties Responsible

Section	Title	Responsible Party	Qualified Persons
	Certificate of Authors	All parties provided input	All
	Signature Pages	All parties provided input	All
1	Summary	All parties provided input	All
2	Introduction	MMTS	Frank Grills, P.Eng.
3	Reliance on Other Experts	MMTS	Frank Grills, P.Eng.
4	Property Description and Location	Discovery	Bill Gilmour, P.Geo.
5	Accessibility, Climate, Local Resource Infrastructure and Physiography	Discovery	Bill Gilmour, P.Geo.
6	History	Discovery	Bill Gilmour, P.Geo.
7	Geological Setting and Mineralization	Discovery	Bill Gilmour, P.Geo.
8	Deposit Types	Discovery	Bill Gilmour, P.Geo.
9	Exploration	Discovery	Bill Gilmour, P.Geo.
10	Drilling	Discovery	Bill Gilmour, P.Geo.
11	Sample Preparation, Analysis, and Security	Discovery	Bill Gilmour, P.Geo.
12	Data Verification	Ginto	Marc Jutras, P.Eng.
13	Mineral Processing and Metallurgical Testing	Ausenco	Paul Staples, P.Eng.
14	Mineral Resource Estimates	Ginto	Marc Jutras, P.Eng.

Section	Title	Responsible Party	Qualified Persons
15	Mineral Reserve Estimates	MMTS	Marc Schulte, P.Eng.
16	Mining Methods	MMTS	Marc Schulte, P.Eng.
17	Recovery Methods	Ausenco	Paul Staples, P.Eng.
18	Infrastructure		
	18.1;18.2	Ausenco	Paul Staples, P.Eng.
	18.3.1;18.3.2;18.3.3;18.3.4;18.3.5;18.3.6;18.3.7; 18.3.8	Knight Piésold	Les Galbraith, P.Eng.
	18.3.9	Linkan Engineering	Sam Billin, P.Eng.
	18.4;18.5	MCA	Malcolm Cameron, P.Eng.
19	Market Studies and Contracts	MMTS	Marc Schulte, P.Eng.
20	Environmental Studies, Permitting and Social or Community Impact (except for Section 20.3)	Knight Piésold	Les Galbraith, P.Eng.
	20.3	pHase	Andrea Samuels, P.Geo.
21	Capital and Operating Costs		
	Table 21-1; Table 21-6; Table 21.9	MMTS. Ausenco, Linkan, Knight Piésold, MCA provided input	All applicable QPs
	21.1.1	MMTS	Frank Grills, P.Eng.
	21.1.2;21.2.1;21.3.1 to 21.3.12	MMTS	Marc Schulte, P.Eng.
	21.1.13;21.2.5;21.3.16	Linkan Engineering	Sam Billin, P.Eng.
	21.1.11;21.1.12;21.2.3;21.3.15	Knight Piésold	Les Galbraith, P.Eng.
	21.1.3;21.1.4;21.1.5;21.1.6;21.1.7;21.1.8;21.1.9; 2.1.10;21.2.2;21.3.13;2.3.14;	Ausenco	Paul Staples, P.Eng.
	21.1.9	MCA	Malcolm Cameron, P.Eng.
22	Economic Analysis	MMTS	Marc Schulte, P.Eng.
23	Adjacent Properties	Discovery	Bill Gilmour, P.Geo.
24	Other Relevant Data and Information	MMTS	Frank Grills, P.Eng.
25	Conclusions and Interpretations	All parties provided input	All
26	Recommendations	All parties provided input	All
27	References	All parties provided input	All

The QPs preparing this Prefeasibility Study are specialists in geology, exploration, mineral resource and mineral reserve estimation and classification, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, infrastructure, capital and operating cost estimation, and mineral economics.

None of the QPs or associates employed in the preparation of this report have any beneficial interest in SMG. The QPs are not insiders, associates, or affiliates of SMG. The results of this technical report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between SMG and the QPs. The QPs are being paid a fee for their work in accordance with normal professional consulting practice.

2.3 Site Visits and Scope of Personal Inspection

Table 2-2 provides a list of QP site visits conducted to the Project site.

Table 2-2 QP Site Visits

QP	Dates	Accompanied By	Description of Inspection
Marc Schulte, P.Eng.	September 12, 2019		Viewed the general topography, inspected proposed pit and stockpile locations, and the locations of existing and proposed infrastructure.
Les Galbraith, P.Eng.	September 12, 2019		Reviewed the general topography and the proposed location of site facilities.
Bill Gilmour, P.Geo.	December 2, 2020		Inspected an active RC drill and observed the drilling and sampling of RC rock chips. Time was spent with the geology team discussing sampling methods, including QC/QA methods.
Paul Staples, P.Eng.	December 8, 2020	Judy Stoeterau (SMG)	Reviewed the proposed mill site and the core.
Andrea Samuels, P.Geo.			Did not visit Site.
Malcolm Cameron, P.Eng.			Did not visit Site.
Frank Grills, P.Eng.			Did not visit Site.
Sam Billin, P.Eng.			Did not visit Site.
Mark Jutras, P.Eng.	November 25, 2020	Larry Yau, Ray Mah	Core logging, core cutting, core storage facilities were inspected. Core logging and sample preparation procedures were examined. Drilling of a reverse circulation hole at site was observed. Geology office was visited. Acquisition and storage procedures of geologic data were examined.

2.4 Effective Dates

There are a few effective dates pertinent to the Report, as follows:

- Effective date for the Mineral Resource estimates: February 3, 2021
- Effective date of Mineral Reserve estimates: March 31, 2021
- Effective date of this NI 43-101 Technical Report: May 10, 2021.

2.5 Information Sources and References

The authors, in writing this Report use sources of information as listed in the references Section 27. Reports have been prepared by qualified persons holding post-secondary geology, or related university degree(s), and are therefore deemed to be accurate.

The purpose of this Report is to provide a NI 43-101 compliant Prefeasibility Study Technical Report for the Project.

The Property is in the Cariboo region of central BC, 6 km east of the community of Likely, and 66 km northeast of the City of Williams Lake. The Property consists of 50 Mineral Titles Online (MTO) mineral claims, of which 20 are legacy claims. These mineral titles form a contiguous block covering an area of approximately 9,319 ha. The Property is 100% owned by SMG; subject to four separate net smelter return (NSR) royalties on some of the mineral tenures.

The main resource, consisting of the Main and North Zones, is located west of the northwest end of Spanish Lake, and is centred at approximate Universal Transverse Mercator (UTM) coordinates 604,425 East and 5,827,900 North (NAD 83, Zone 10). It is located mainly within mineral claim 204667 and mineral claims 204225 and 204226.

Data derived from engineering companies, consultants, and authors are listed in Section 27.

2.6 Units, Currency, and Rounding

Unless otherwise specified or noted, the units used in this technical report are metric. Every effort has been made to clearly display the appropriate units being used throughout this technical report. Currency is in Canadian dollars ($) unless otherwise noted.

This NI 43-101 report includes technical information that required subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QP does not consider them to be material.

2.7 Previous Technical Reports

Refer to Section 27.

2.8 Important Notice

Each QP that assisted in the preparation of this Prefeasibility Study assumes responsibility for those sections or areas of this report that are referenced in Table 2-1. None of the QPs; however, accepts any responsibility or liability for the sections or areas of this report that were prepared by other QPs.

This NI 43-101 was prepared to allow SMG to reach informed decisions respecting the development of the SMG. Except for the purposes legislated under provincial securities law: (a) any use of this report by any third party is at that party's sole risk, and none of the QPs (nor any of the companies for whom they work) shall have any liability to any third party for any such use for any reason whatsoever, including negligence, and (b) each of the QPs hereby disclaims responsibility for any indirect or consequential loss arising from any use of this report or the information contained herein. This report is intended to be read as a whole, and sections should not be read or relied upon out of context. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

This report contains estimates, projections, and conclusions that are forward-looking information within the meaning of applicable securities laws. Forward-looking statements are based upon the responsible QP’s opinion at the time that they are made, but in most cases involve significant risk and uncertainty. Although each of the responsible QPs has attempted to identify factors that could cause actual events or results to differ materially from those described in this report, there may be other factors that cause events or results to not be as anticipated, estimated, or projected. None of the QPs undertake any obligation to update the forward-looking information.

This report contains the expression of the professional opinion of the QPs based on: (i) information available at the time of preparation, (ii) data supplied by outside sources, (iii) conclusions of other technical specialists named in this report, and (iv) the assumptions, conditions, and qualifications in this report. The quality of the information, conclusions, and estimates contained herein are based on industry standards for engineering and evaluation of a mineral project and are consistent with the intended level of accuracy for a Prefeasibility Study.

As permitted by Item 12 of Form 43-101F1, the QPs have, in the preparation of this Report, relied upon certain data provided to the QPs by the Owner and certain other parties. The relevant data and the extent of reliance on such data are described in Section 3 of this Report.

3 RELIANCE ON OTHER EXPERTS

The following Non-QP specialist was relied upon for specific advice:

- Mr. Jordan Ross (PricewaterhouseCoopers) - post tax analysis of financial model, including Canadian Federal corporate income tax, BC Provincial income tax, and BC Mineral tax.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Property is in the Cariboo region of central British Columbia, approximately 6 km southeast of Likely and 66 km northeast of Williams Lake, as shown in Figure 4-1. The Property, with a general northwest-southeast orientation, is situated between Quesnel Lake and Spanish Lake. The property ranges from UTM coordinates 599,000 East (Datum NAD83, Zone 10) to 613,600 East, and from 5,817,800 North to 5,832,000 North.

The Resource, within the Main and North Zones, is located west of the northwest end of Spanish Lake and is centred at approximate UTM coordinates 604,400 East and 5,827,900 North. It is located mainly within the mineral title 204667, as well as mineral titles 201021 and 204226.

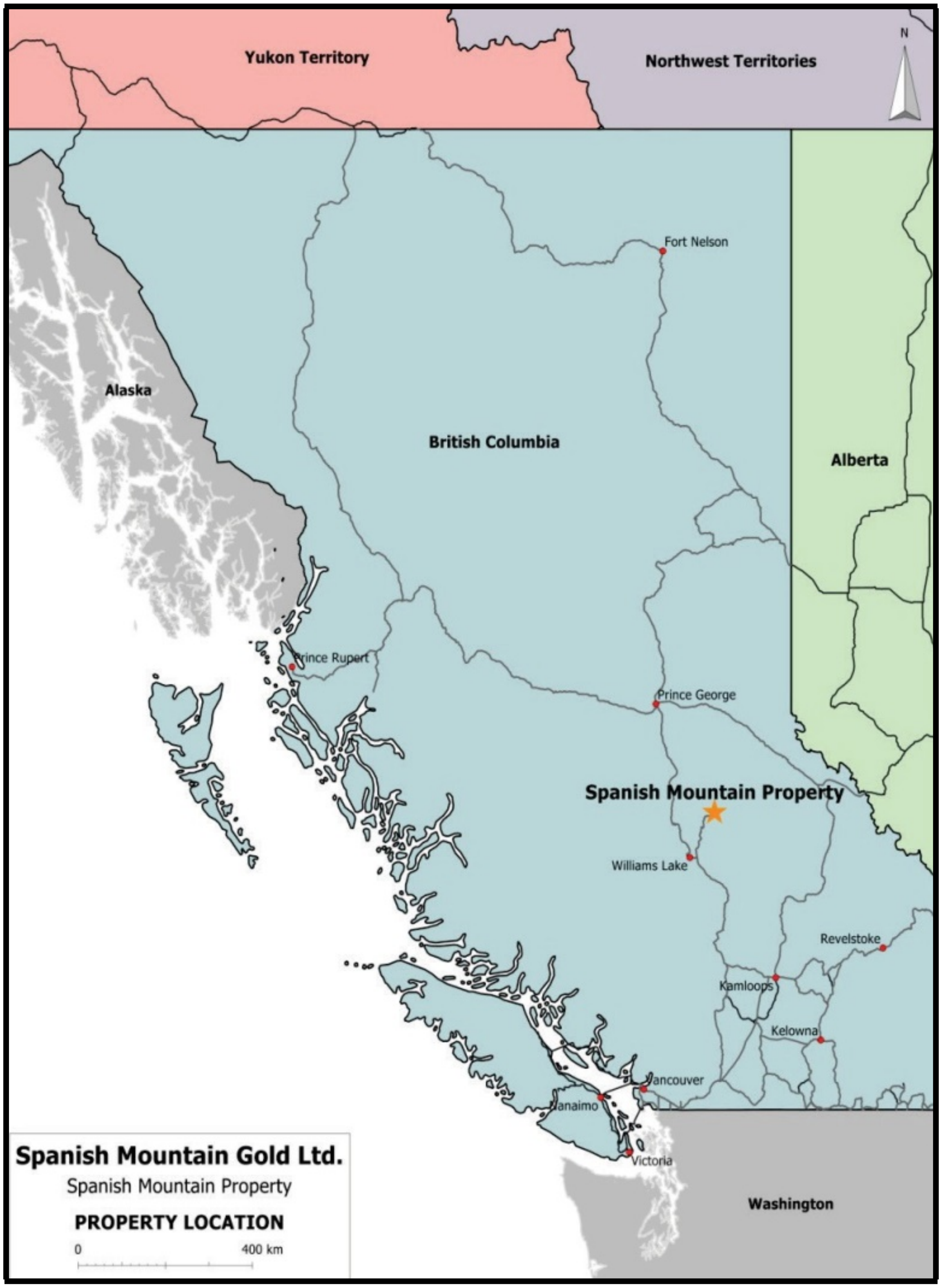


Figure 4-1 Property Location

4.2 Description

The Property consists of 50 MTO mineral titles, of which 20 are legacy claims. These mineral titles form a contiguous block covering an area of approximately 9,319 ha. This is smaller than the area on which SMG pays assessment fees (10,335 ha), as some MTO claims overlie legacy claims, either those of SMG or those of a third party.

Third party ownership overlying the Property comprises (Figure 4-2):

- district lots of several private homeowners along the eastern side of Quesnel Lake and one small, isolated parcel (DL12083) at the northwest end of Spanish Lake;
- third parties own 121 placer claims and four leases (totalling 4,440 ha) overlying the Property.

The mineral titles lie on British Columbia Mineral TRIM Map Sheets 093A.053, 054 and 063. All titles are 100% owned by SMG. Table 4-1 lists the details of the titles. SMG also owns eight overlying placer titles (2,004 ha) in the area (Figure 4-3).

A reason for the abundance of placer claims/leases is that much of the area of the Property is in designated placer areas (ID 329583 and 330210).

Cedar Point Provincial Park is a small 8-hectare Class C park, located where Cedar Creek enters Quesnel Lake. Part of the Park underlies claim 517485.

Table 4-1 Mineral Title Description

Tenure Number	Claim Name	Area (ha)	Map Number	Registered Owner	Good To Date**
204021	PESO	225.00	093A.053	Spanish Mountain Gold Ltd.	2030/Feb/27
204224	DON 1	25.00	093A.053	"	2030/Feb/27
204225	DON 2	25.00	093A.053	"	2030/Feb/27
204226	DON 3	25.00	093A.053	"	2030/Feb/27
204227	DON 4	25.00	093A.053/063	"	2030/Feb/27
204274	MARCH 1	500.00	093A.053/063	"	2030/Feb/27
204275	MARCH 2	100.00	093A.053/063	"	2030/Feb/27
204334	JUL 2	225.00	093A.053/063	"	2030/Feb/27
204667*	CPW	100.00	093A.053	"	2030/Feb/27
205151	MEY 1	500.00	093A.053/063	"	2030/Feb/27
373355	ARMADA	450.00	093A.053	"	2030/Feb/27
373415	N.R.1	25.00	093A.053	"	2030/Feb/27
399410	ARMADA 2	500.00	093A.053	"	2030/Feb/27
399411	ARMADA 4	500.00	093A.053	"	2030/Feb/27
399412	ARMADA 5	500.00	093A.053	"	2030/Feb/27
399413	ARMADA 6	25.00	093A.053	"	2030/Feb/27
399415	ARMADA 8	25.00	093A.053	"	2030/Feb/27
399417	ARMADA 10	25.00	093A.053	"	2030/Feb/27
399419	ARMADA 12	25.00	093A.053	"	2030/Feb/27
404303	AG 2	25.00	093A.053	"	2030/Feb/27
502372	SPANISH 1	491.33	093A.053/054	"	2030/Feb/27

Tenure Number	Claim Name	Area (ha)	Map Number	Registered Owner	Good To Date**
502608	SPANISH 2	157.23	093A.053/054	"	2030/Feb/27
503338	SPANISH 3	196.58	093A.053/054	"	2030/Feb/27
510115	GOLDEN AIRPORT	274.82	093A.063	"	2030/Feb/27
512541		117.89	093A.053	"	2030/Feb/27
512542		78.58	093A.053	"	2030/Feb/27
512544		78.58	093A.053	"	2030/Feb/27
512547		19.65	093A.053	"	2030/Feb/27
512549		78.58	093A.053	"	2030/Feb/27
512572	FISCHER CREEK	196.34	093A.063	"	2030/Feb/27
514947	GOLD TREND	117.76	093A.063	"	2030/Feb/27
517007	GOLD	19.64	093A.063	"	2030/Feb/27
517056	GOLDIE	58.90	093A.063	"	2030/Feb/27
517098	GOLD3	39.26	093A.063	"	2030/Feb/27
517446		19.65	093A.053	"	2030/Feb/27
517485		1335.78	093A.053	"	2030/Feb/27
521302	AKV	58.94	093A.053	"	2030/Feb/27
537371	MOOREHEAD 12	78.52	093A.063	"	2030/Feb/27
537372	MOOREHEAD 13	39.27	093A.063	"	2030/Feb/27
538658	MOOREHEAD 14	117.86	093A.053	"	2030/Feb/27
603743	LIKELY GULCH	78.52	093A.063	"	2030/Feb/27
810602	SPAN 3	19.63	093A.063	"	2030/Feb/27
822682 Δ		78.56	093A.053	"	2030/Feb/27
844711	SPAN 4	19.63	093A.063	"	2030/Feb/27
849064	SPAN 5	472.05	093A	"	2029/Jul/01
849066	SPAN 6	472.06	093A	"	2029/Jul/01
849069	SPAN 7	491.71	093A	"	2029/Jul/01
849070	SPAN 8	491.96	093A	"	2029/Jul/01
1062098	SPANISH MOUNTAIN SOUTH	786.48	093A	"	2029/Aug/01
1071029	SPAN SW	19.65	093A	"	2020/Sep/13
Total:		10355.41			

*Claims in **red** are subject to the Mickle option agreement*
*Claim in **blue** is subject to the Wallster and McMillan option agreement*
*Claims in **green** are subject to the Cedar Creek option agreement*
*Claims in **purple** are subject to the Acrex purchase agreement*
* *Claim on which work was done*
Δ *Claim 822682 is converted from legacy claim 204727, which is subject to the Mickle option agreement*

The locations of the claims are shown in Figure 4-2.

Note that due to Covid-19, the Mineral Titles Branch has issued a Title Protection Event, granting an Extension of Time until December 31, 2021. This Event applies to title 1071029 and to SMG's eight placer claims. SMG has until this date to complete and file assessment on the titles.

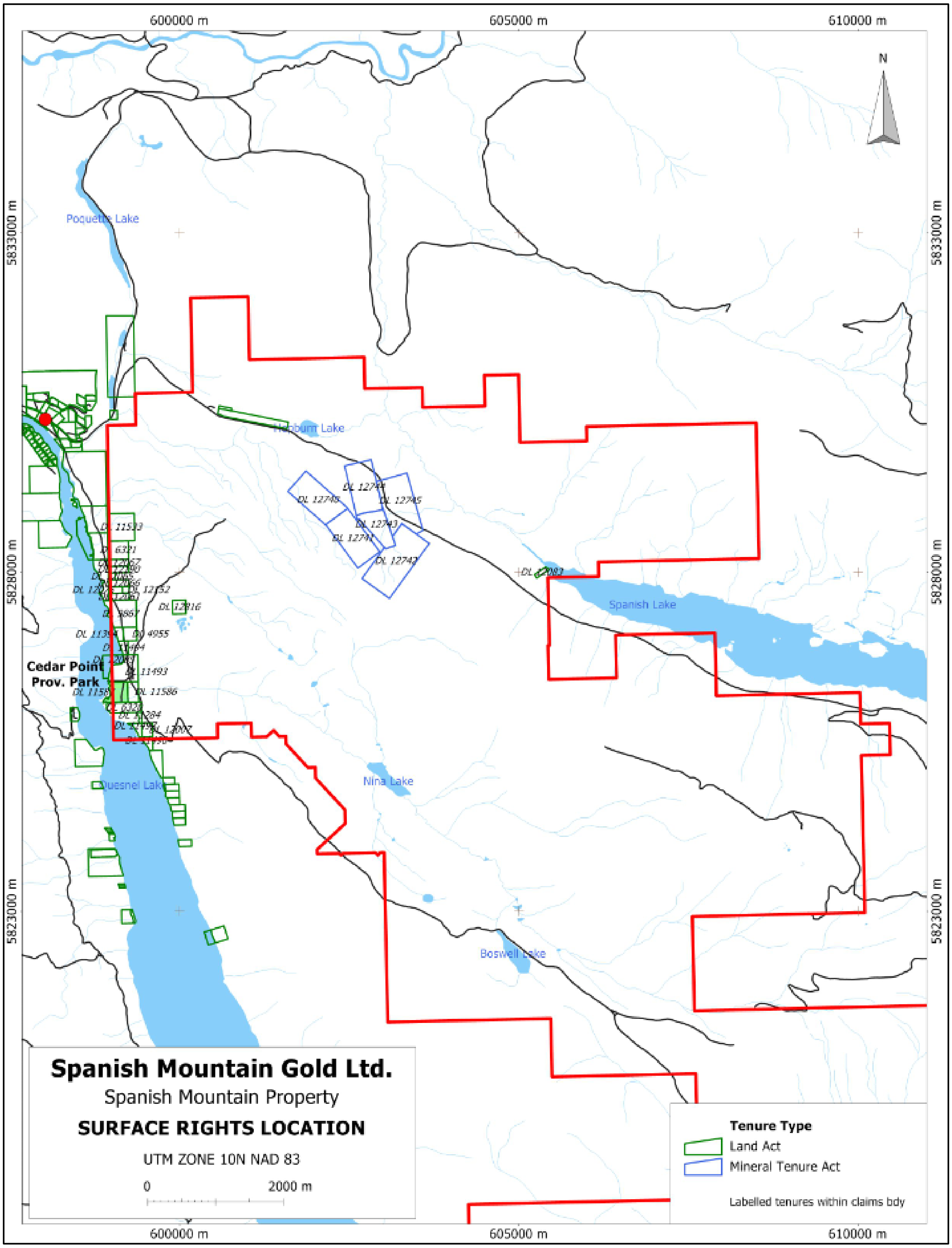


Figure 4-2 Surface Rights Locations

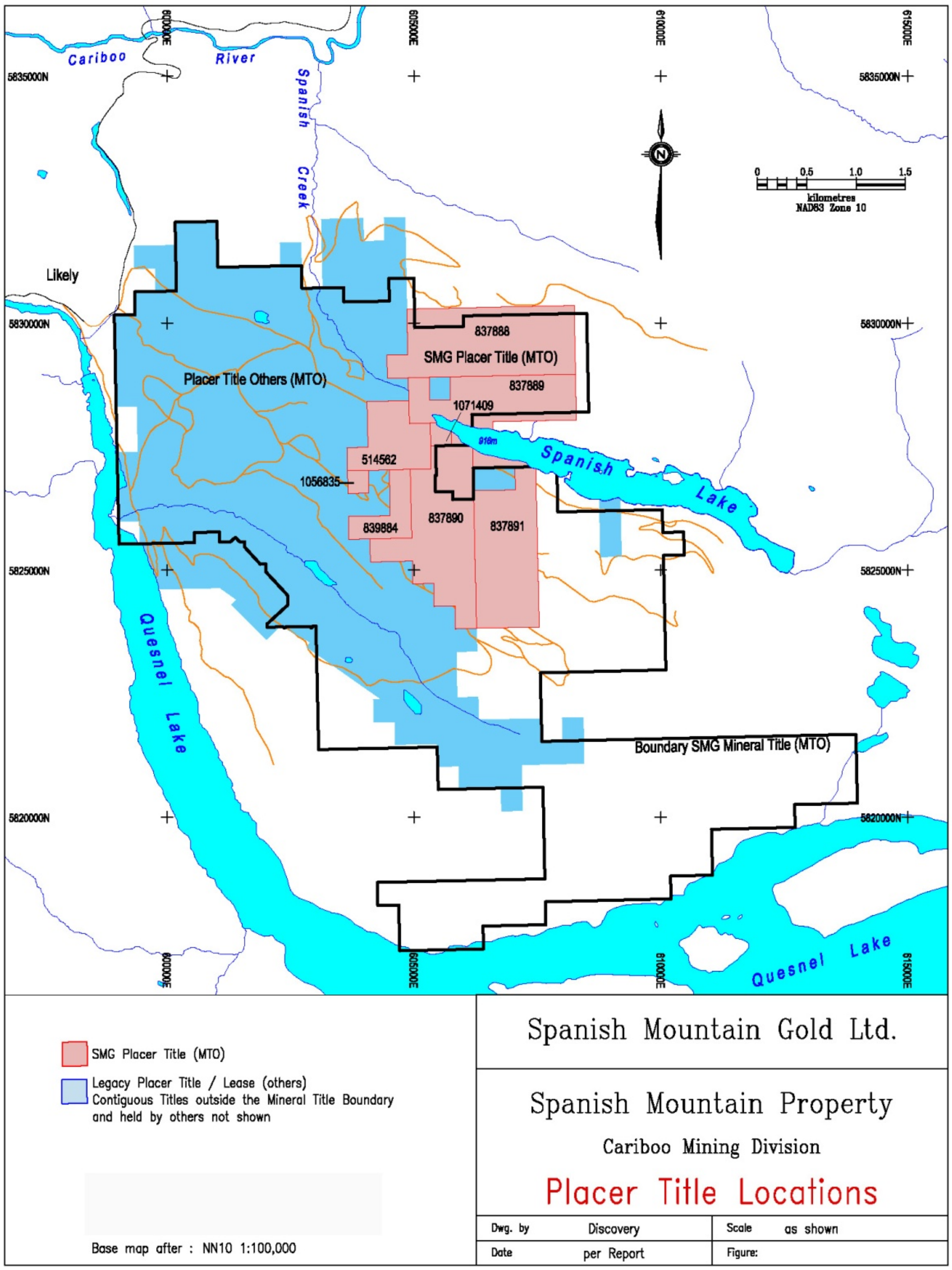


Figure 4-3 Placer Claim Locations

4.3 Ownership

Spanish Mountain Gold ("SMG"), with offices at 1120 – 1095 West Pender Street, Vancouver, BC, owns all 50 mineral titles comprising the Property. The company was formerly named Skygold Ventures Ltd, with the change in name effective January 14, 2010. Four underlying option agreements pertain to a certain number of the mineral titles:

1. A 2.5% net smelter return ("NSR") royalty payable to Robert E. Mickle ("Mickle") on 12 mineral titles
2. A 2.5% NSR royalty payable to D.E. Wallster ("Wallster") and J.P. McMillan ("McMillan") on one mineral title
3. A 2.5% NSR royalty payable to G. Richmond ("Richmond") on two mineral titles
4. A 4% NSR royalty payable to Acrex Ventures Ltd on 11 mineral titles

Details of the first underlying agreement with R.E. Mickle are as follows:

- An option agreement dated January 10, 2003, between Wildrose Resources Ltd ("Wildrose") and Mickle, of Likely, BC, for Wildrose to earn a 100% interest in 12 mineral titles as listed in Table 4-1 (coloured in red). Wildrose is a wholly owned subsidiary of SMG. The agreement provides for escalating cash payments totalling $100,000 over five years. These payments have all been made. There is provision for a 2.5% NSR royalty payable to Mickle for any production from these claims, of which 1.5% may be purchased by payment of $500,000 to Mickle.

Details of the second underlying agreement with Wallster and McMillan are as follows:

- An option agreement dated January 20, 2003, between Wildrose (the Optionee), SMG (the Assignee), and Wallster as to a two-thirds interest and McMillan as to a one-third interest, (Wallster and McMillan being referred to collectively as the Underlyers), for the Optionee and the Assignee to earn a 100% interest in the 204667-mineral title. The agreement provides for escalating cash and/or shares of equal value payments totalling $348,000 over nine years, in addition to 30,000 common shares of the Assignee on signing. These obligations have been met. There is a provision for a 2.5% NSR royalty payable to the Underlyers for any production from the 204667-mineral title, of which 1% may be purchased by payment of $500,000 to the Underlyers at the commencement of commercial production from the mineral title.

On January 20, 2003, Wildrose and SMG entered into an option agreement under which SMG could earn a 70% interest in the Property, including those mineral titles included in the two agreements above. Under this agreement, SMG was obligated to complete $700,000 in exploration expenditures on the Property, issue to Wildrose 200,000 common shares of SMG and a further consideration of cash and/or shares valued at $200,000 and satisfy underlying agreement terms. On March 29, 2005, SMG advised Wildrose that it had fulfilled its option requirements to earn its interest, and a joint venture was created, of which SMG was the operator.

On November 30, 2007, SMG entered into a letter agreement, whereby SMG would acquire all the issued and outstanding shares of Wildrose in exchange for common shares of SMG by way of a Plan of Arrangement under the British Columbia Corporations Act (the "Transaction").

Under the proposed Transaction, Wildrose shareholders would receive 0.82 common shares of SMG for each common share of Wildrose. SMG would assume outstanding warrants and stock options of Wildrose on the basis that each warrant or option of Wildrose would be exchanged for 0.82 of one warrant or option and the exercise price of such warrant or option would be appropriately adjusted in accordance with the exchange ratio. On July 9, 2008, SMG announced that "… all the conditions to the acquisition by Spanish Mountain Gold of Wildrose Resources Ltd. pursuant to a plan of arrangement under the Business Corporations Act (British Columbia), have been satisfied and the acquisition has now been completed." By virtue of the merger, SMG became responsible for the underlying agreements.

Details of the third underlying agreement on the Cedar Creek mineral titles with Cedar Mountain Exploration Inc. ("Cedar Mountain") are as follows:

- A purchase agreement dated June 15, 2010, between SMG and Cedar Mountain, for SMG to earn a 100% interest in two mineral titles as listed in Table 4-1. The agreement provided for a cash payment totalling $500,000 on signing. There is provision for a 2.5% NSR royalty payable to Richmond for any production from these titles, which may be purchased by SMG through the payment to Richmond of $500,000 per 1% NSR.

Details of the fourth underlying agreement on the Acrex mineral titles with Acrex Ventures Ltd ("Acrex") are as follows:

- A purchase agreement dated July 25, 2012, between SMG and Acrex, for SMG to earn a 100% interest in 11 mineral titles as listed in Table 4-1. The agreement provided for a cash payment totalling $500,000 on signing and the issuance of 2,000,000 common shares of SMG. In addition, SMG granted and assumed a third-party royalty such that the Acrex titles are subject to a 4% NSR, which may be purchased by paying $2,000,000 at any time after commencement of commercial production.

4.4 Permits and Liabilities

A multi-year Mines Act Permit (MX-10-199) has been issued for the Property by the BC Ministry of Energy and Mines. Reclamation bonds for the Property totalling $85,000 are held in trust by the British Columbia Government, to cover the cost of reclamation on the Property. Since the project is ongoing, the bonds remain outstanding. Refer to Section 20 for environmental liabilities and permits.

4.5 Comments on Section 4

There are 4,440 ha of placer claims and leases that belong to third parties and which overlie in whole or in part the Property mineral titles (Figure 4-4). They may become an issue when planning the locations of mine dumps, tailings, and other infrastructure. When converting from a placer claim to a placer mining lease, or from a mineral title to a mining lease, the older of non-lease titles have priority.

The Resource underlies areas on which royalties are payable, which may or may not significantly affect the economic potential of the Property. Other than these issues, to the extent known, there are no significant factors and risks that may affect access, title or right or ability to perform work on the Project.

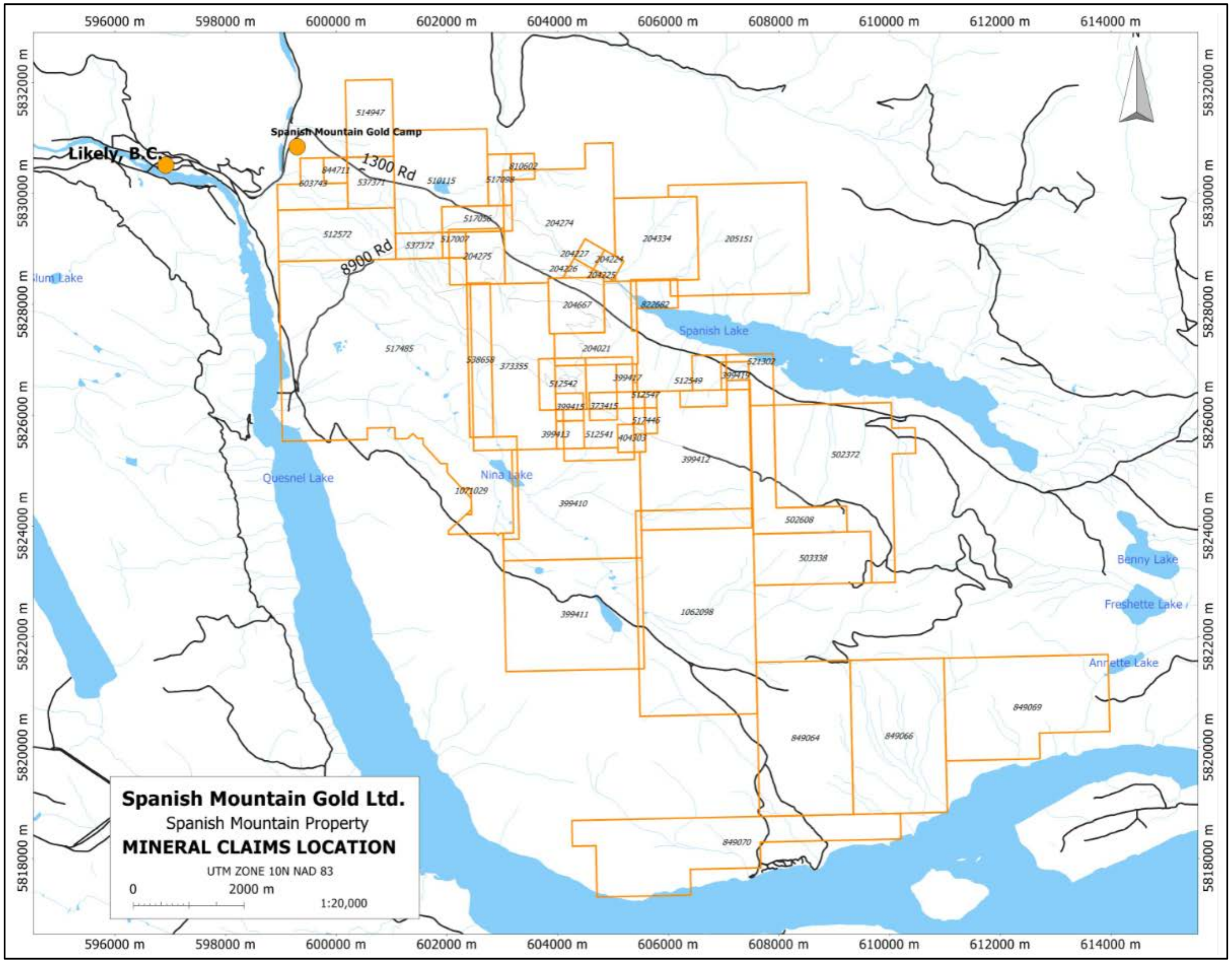


Figure 4-4 Mineral Title Locations

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

5.1 Access

The Property can be reached from Williams Lake via a paved secondary road that leaves Highway 97 at 150 Mile House, approximately 16 km south of Williams Lake, and continues for 87 km to Likely (Figure 5-1). From Likely, the central and northern part of the Property is accessed from FSR 1300, which begins east of Likely and continues through the centre of the Property. The southern portion of the Property is accessed from Likely along the Cedar Creek / Winkley Creek Road (FSR 3900), for about 10 km. Numerous logging roads lie throughout the Property and offer good access to most areas. A gravel airstrip is located along the 1300 FSR between kilometres 2 and 3.

5.2 Climate

The climate of the Likely area is modified continental with cold snowy winters and warm summers. Likely has annual average precipitation of approximately 70 cm. Snowfall on the Property is commonly about 200 cm between the months of October and April. Most small drainages tend to dry up in the late summer. Drilling programs can be conducted on a year-round basis.

5.3 Local Resources

SMG has a modern, full-service facility on purchased land near the Property that provides a base for operations. Likely has basic amenities including a motel, hotel, rental cabins, corner store, gas pumps, and a seasonal restaurant. Some heavy equipment is also available for hire from local contractors. All services and supplies are readily available in Williams Lake, an hour's drive from Likely. The Williams Lake airport is serviced by Central Mountain Air and Pacific Coastal Airlines, which provide daily service with Vancouver, BC, and by Air Canada, which provides less frequent service.

5.4 Infrastructure

The main access area to the Property is the Likely Road, which passes north of the access road to the Mount Polley copper-gold mine, owned by Imperial Metals Ltd. This mine is situated about 15 km southwest of the centre of the Property. Power is available at Likely, with a major line in place to Mount Polley. The mine will receive power from a new 138kV transmission line. BC Hydro will establish a new 138 kV Substation near Highway 97, to be known as SMMX Substation. The new substation will be fed at 230 kV from existing BC Hydro line 2L95. The new substation will contain a 30 MVA step-down transformer to 138 kV, and metering at 138 kV. A set of outgoing suspension insulators will form the Point of Interconnection (POI). The new substation will be owned and operated by BC Hydro. Everything downstream of the POI will be owned and operated by Spanish Mountain. A new 138 kV receiving substation will be constructed on-site.

Water is abundant in the area. For further details on water sources refer to Section 18.

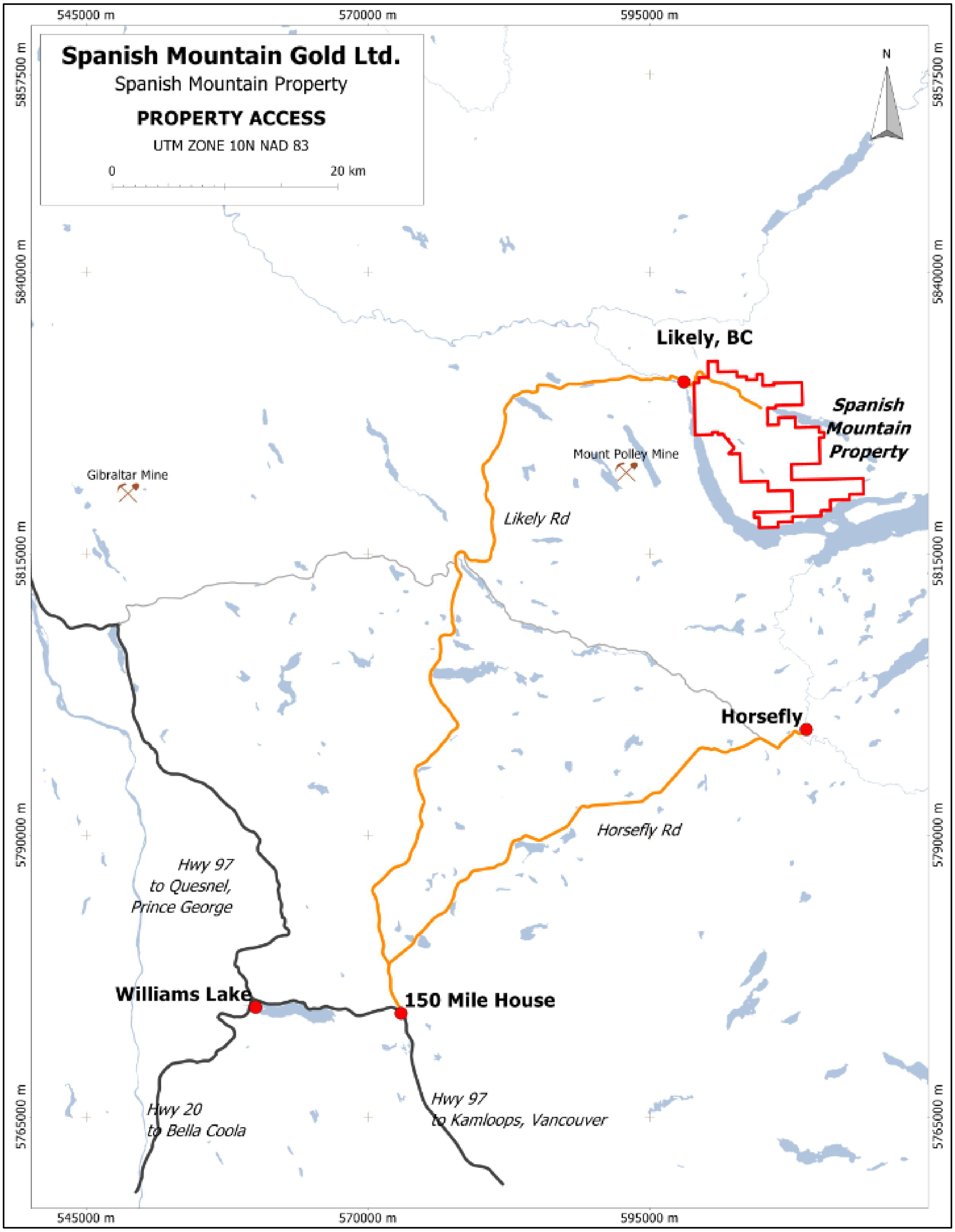


Figure 5-1 Property Access

5.5 Physiography

The Property covers an area of up to approximately 14 km north to south by 15 km east to west, situated between Spanish Lake on the east and Quesnel Lake on the west. Physiographically, the area is situated within the Quesnel Highland, which is transitional between the gently undulating topography of the Cariboo Plateau to the west, and the steeper, sub-alpine to alpine terrain of the Cariboo Mountains to the east. The terrain is moderately mountainous with rounded ridge tops and U-shaped valleys. Topography is locally rugged with occasional cliffs and moderately incised creek valleys. Within the Property, elevations range from 910 metres above sea level ("masl") at Spanish Lake to 1,587 masl at the Peak of Spanish Mountain. Drainage is via Spanish Creek, which drains to the northwest into Cariboo Creek, and via Cedar Creek, which drains to the west into Quesnel Lake. Quesnel Lake flows into Quesnel River, and, joined by Cariboo Creek, flows westerly to eventually join the Fraser River near the town of Quesnel.

Overburden depths are quite variable, ranging from one to ten metres in most of the Main Zone, to over 70 m further west in the Cedar Creek area. During the last glacial period, the ice advanced in a northwesterly direction (Tipper, 1971). Rock outcroppings are scarce and are typically found along the crest of ridges, in incised river and creek gullies, and along shorelines.

Vegetation in the area consists of hemlock, balsam, cedar, fir, and cottonwood found in valley bottoms and spruce, with fir and pine at higher elevations. Alder, willow, and devil's club grow as part of the underbrush, which can be locally thick. Parts of the Property have been logged at various times, resulting in areas having open hillsides with younger forest growth. In addition, large sections of the pine forest have recently been affected by mountain pine beetle infestation.

5.6 Comments on Section 5

There is sufficient land available within the mineral tenure held by SMG for tailings storage, mine waste storage, the process plant, and related site infrastructure.

Existing power and water sources, manpower availability, and transport options indicate that there are reasonable expectations that sufficient labour and infrastructure will continue to be available to support declaration of Mineral Resources and the proposed life-of-mine (LOM) plans.

6 HISTORY

The earlier history of the Property has been summarized by Page (2003), and by Singh (2008). Table 6-1 gives a summary of the historical work, up to and including 2004, in tabular form, and has been adapted from Singh (2008) with minor edits. The 2005 to 2009 exploration programs carried out by SMG at that time were done under its former name of Skygold Ventures Ltd. Work conducted from 2005 to the present is described in more detail in Sections 10 and 11 of this Report.

6.1 Exploration and Development History

The main events of the SMG's history prior to Spanish Mountain Gold's involvement are summarized in Table 6-1.

Table 6-1 Summary of Historical Information

Year	Company	Work Done
2004	Wildrose Resources Ltd	2,506 m of RC drilling in 34 holes, 2,419 m of trenching, soil sampling
2003	Wildrose Resources Ltd	30-line km of grid. IP survey (23-line km), soil sampling (1,479 samples), geological mapping. Spanish Mountain options the Property and begins funding exploration
2002	Wildrose Resources Ltd	Small geochemical sampling program
1999-2000	Imperial Metals Ltd.	Imperial Metals options the Property and attempts bulk samples from five pits. From one pit, a 1,908-tonne bulk sample (screened portion of 6,000 tonnes) averages 3.02 g/t Au, based on sampling of 64 truckloads. Blast hole drilling (201 samples from 182 holes) averaged 2.20 g/t Au, based on assays performed at Mount Polley
1996	Cyprus Resources Ltd.	2,590 m of trenching signifying the first effort to explore for bulk mineable type disseminated gold mineralization. 230 m of trench TR96-101 assayed 0.745 g/t Au.
1995	Eastfield Resources Ltd.	Optioned the Property to Consolidated Logan Mines who then optioned it to Cyprus Resources Ltd.
1993-1994	Cogema Canada Ltd.	30 trenches with 900 rock/channel samples
1992	Renoble Holdings Inc.	Stockpiled 635 tonnes from a small open pit in the Madre zone ("High Grade zone"). The material was processed in two mill runs; 318 tonnes were sent to the Premier Mill (46 troy ounces recovered) and 105 tonnes were sent to the Bow Mines Mill (Greenwood, BC) with 105 troy ounces recovered
1992	Eastfield Resources Ltd.	Consolidated the Spanish Mountain property
1986-1988	Pundata Gold Corp.	37 core drillholes (3,273 m), 15 RC holes (1,237 m), 848 m of trenching, geological mapping, sampling (5,350 samples), metallurgical testing of 11 samples, preliminary resource estimate
1987	Placer Dome Inc.	Optioned north and west and south areas of the Property. 7 percussion holes (338 m) were drilled: 5 along the northwest ridge of Spanish Mountain and 2 near the Cedar Creek drainage. Significant gold values were obtained from overburden section of several holes
1986	Mandusa Resources Ltd.	Optioned the north and southern areas of the Property. Conducted geological mapping and IP surveys, and drilled 6 percussion holes (357 m)
1985	Mt. Calvery Resources Ltd.	Phase 1: 600 m of trenching and sampling, 7 RC holes (655 m). Phase 2: 820 m of backhoe trenching (550 1-m channel samples), 29 RC holes (2,521 m). A preliminary resource estimate was made. Phase 3: 7 core drillholes. Teck Corp. provided funding for Phases 2 and 3
1984	Mt. Calvery Resources Ltd.	Prospecting, geological mapping, rock, and soil sampling. 2,225 m of trenching, 10 core drillholes (467 m), 10 RC holes (589 m)

Year	Company	Work Done
1983	Whitecap Energy Inc.	Soil sampling (409 samples) on the CPW claim with values up to 5,100 ppb Au. 100 m of trenching in 3 trenches
1983	Lacana Mining Corp.	Prospecting identified strong gold anomalies coincident with silicified argillite north of Spanish Lake
1982	C.P. Wallster	Staked the CPW claim, as the Mariner II claim had lapsed earlier that year
1981	Aquarius Resources Ltd.	Soil sampling, airborne geophysical EM survey
1979, 1980 and 1982	E. Schultz, P. Kutney and R.E. Mickle	Prospecting, sampling, stripping by D-7 and D-8 cats. 240 m of trenching. Little information is available for this work
1979	Aquarius Resources Ltd.	Surface exploration and regional assessment of the Likely area
1977-1978	LongBar Minerals	Prospecting (14 rock samples), geological mapping, soil sampling (60 samples) and trenching (14 trenches)
1976	M.B. Neilson	Staked the Mariner II claim (“High grade zone”). A few samples were collected
1971	Spanallan Mining Ltd.	Magnetometer survey on the Cedar Creek drainage
1947	El Toro BC Mines	8 drillholes (792 m), 4 tons of handpicked ore shipped to the Tacoma Smelter
1938	N.A. Timmins Corp.	Overburden stripping, drove 2 small adits on large quartz veins
1933	Dickson and Bailey	Gold discovered in quartz veins on the northwest flank of Spanish Mountain at 1100 m elevation
1921		Placer gold discovered in bench deposits on Cedar Creek

6.2 Previous Mineral Resource Estimates

The following Mineral Resource Estimates are from 2011, 2012, 2014, 2017 and 2019. These are not current and are historic in nature and they should not be relied upon.

- November 2011:
    - Measured plus Indicated Resource of 138,030,000 tonnes grading 0.49 g/t gold and 0.64 g/t silver at a 0.20 g/t gold cut-off
    - Inferred Resource of 339,630,000 tonnes grading 0.37 g/t gold and 0.65 g/t silver at 0.20 g/t gold cut-off
- August 2012:
    - Measured plus Indicated Resource of 216,220,000 tonnes grading 0.46 g/t gold and 0.68 g/t silver at a 0.20 g/t gold cut-off
    - Inferred Resource of 316,740,000 tonnes grading 0.36 g/t gold and 0.65 g/t silver at a 0.20 g/t gold cut-off
- April 2014:
    - Measured and Indicated Resource of 237,830,000 tonnes grading 0.46 g/t gold and 0.69 g/t silver at a 0.20 g/t gold cut-off
    - Inferred Resource of 310,970,000 tonnes, grading 0.35 g/t gold and 0.63 g/t silver with a 0.20 g/t cut-off

- May 2017:
    - Measured plus Indicated Resource of 306,530,000 tonnes grading 0.39 g/t gold and 0.64 g/t silver at a 0.15 g/t gold cut-off
    - Inferred Resource of 450,640,000 tonnes grading 0.28 g/t gold and 0.61 g/t silver at 0.15 g/t gold cut-off
- December 2019:
    - Measured plus Indicated Resource of 273,200,00 tonnes grading 0.47 g/t gold and 0.71 g/t silver at a 0.15 g/t gold cut-off
    - Inferred Resources of 52,400,000 tonnes grading 0.37 g/t gold and 0.67 g/t silver at a 0.15 g/t gold cut-off

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Property lies within the Quesnel Terrane of the Intermontane Belt. The rocks of the Quesnel Terrane are predominantly sedimentary and volcanic rocks of the middle to upper Triassic Nicola Group, representing an island arc and marginal basin assemblage. East of the Property, the regional, southwesterly dipping Eureka Thrust marks the western extent of pre-Quesnel Terrane rocks; notably the intensely deformed, variably metamorphosed Proterozoic and Paleozoic pericratonic rocks of the Snowshoe Group. This region also includes the Crooked Amphibolite unit of the Slide Mountain Terrane, of Carboniferous to Permian age, which overlies the rocks of the Snowshoe Group in thrust fault contact, and Quesnel Lake gneiss, of Late Devonian to Carboniferous age.

The stratigraphy of the Quesnel Terrane in the Spanish Mountain area has been examined by Campbell (1978), Struik (1983, 1988), Bloodgood (1988), and more recently by Schiarizza (2016, 2017, 2018). Panteleyev et al. (1996) produced a geological compilation of the Quesnel River - Horsefly area. The Quesnel Terrane in the region consists mainly of a sedimentary package of black graphitic argillites, phyllitic siltstones, sandstones, limestones, and banded tuffs of the Nicola Group, and is weakly metamorphosed. The age of the Nicola Group, based on conodont fossils found south of Quesnel Lake, is Middle to Late Triassic.

Schiarizza (2018) subdivided the Nicola Group rocks in the Spanish Mountain area into three assemblages, two of which occur on the Property. Assemblage One, of Middle Triassic age, consists of siltstone and argillite with lenses of pillowed basalt and volcanic sandstone. These rocks form a northwest trending belt that dips steeply to the southwest and is stratigraphically overlain by Late Triassic Nicola Group Assemblage Two, which comprises volcanic sandstone, conglomerate, and siltstone.

The overlying Nicola Group volcanic rocks of Assemblage Three are in depositional contact with the metasedimentary rocks of Assemblage Two. This unit consists of pyroxene-phyric basalt, pillowed basalt, and basalt breccia, and is exposed in the southwest part of the map area.

In addition, Schiarizza (2016, 2017) re-assigned what were Nicola Group rocks north of Spanish Lake to the middle to upper Triassic Slocan Group. An inferred fault under Spanish Lake and along Spanish Creek marks the new boundary between these units. These two units are of the same age, trend to the northwest, and have very similar lithologies, except for volcaniclastic sediments being restricted to the Nicola Group rocks. However, the structural domains differ. The eastern domain of Slocan Group and underlying Paleozoic rocks is represented by a series of northeast verging folds, cut by younger southwest verging structures. In contrast, the western Nicola Group assemblages are part of the forelimb of a major southwest-verging fold (Schiarizza, 2018).

Figure 7-1 shows the regional geology, based on the work by Schiarizza (2016, 2017). Note that the claim boundary is simplified.

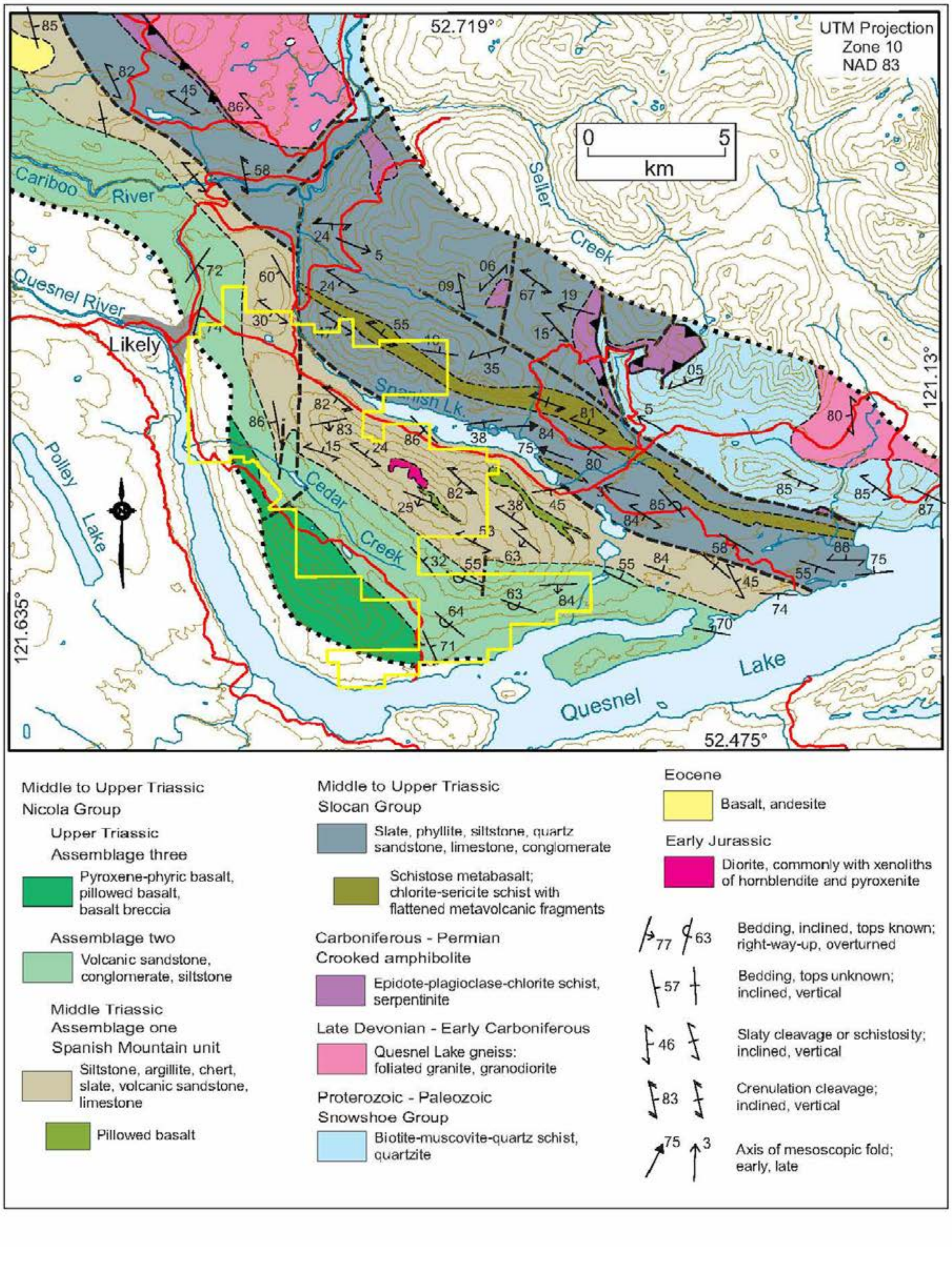


Figure 7-1 Regional Geology (Claims Boundary shown in yellow)

7.2 Property Geology

Much of the information on the Property geology (Figure 7-2) has been taken from Singh (2008). The SMG deposit is within metasediments of the Quesnel Terrane, and is hosted by the phyllite package of rocks, which comprises interbedded slatey to phyllitic, dark grey to black siltstone, carbonaceous mudstone, greywacke, tuff, and minor conglomerate. The main host of the gold mineralization is black, graphitic phyllitic argillite. The sedimentary units have been intruded by plagioclase-quartz-hornblende sills and dykes, which range in thickness from tens of centimetres to as much as 100 m. The intrusions have also been affected by phases of folding, alteration, and quartz veining.

As discussed above, some of the Nicola sedimentary rocks have been reassigned to the Slocan Group. The rocks north of Spanish Lake and Spanish Creek, mostly mapped as a siltstone-argillite unit are now Slocan Group.

The SMG deposit is a bulk-tonnage gold system of finely disseminated gold within black argillites and siltstones, which also contains local high-grade, gold-bearing quartz veins within siltstones, greywackes, and tuff. The largest zone carrying significant gold mineralization is called the Main Zone, which has been traced by drilling over a length of approximately 900 m north-south and a width of 800 m. The stratigraphy of the North Zone is less well understood, but consists of argillites, siltstones and lesser mafic volcanic dykes and sills, covering an area of about 400 m north-south, with a similar width as the Main Zone. The boundary between the North and Main Zones is roughly defined by the 1300 FSR and is underlain by silicified siltstones with mafic dykes.

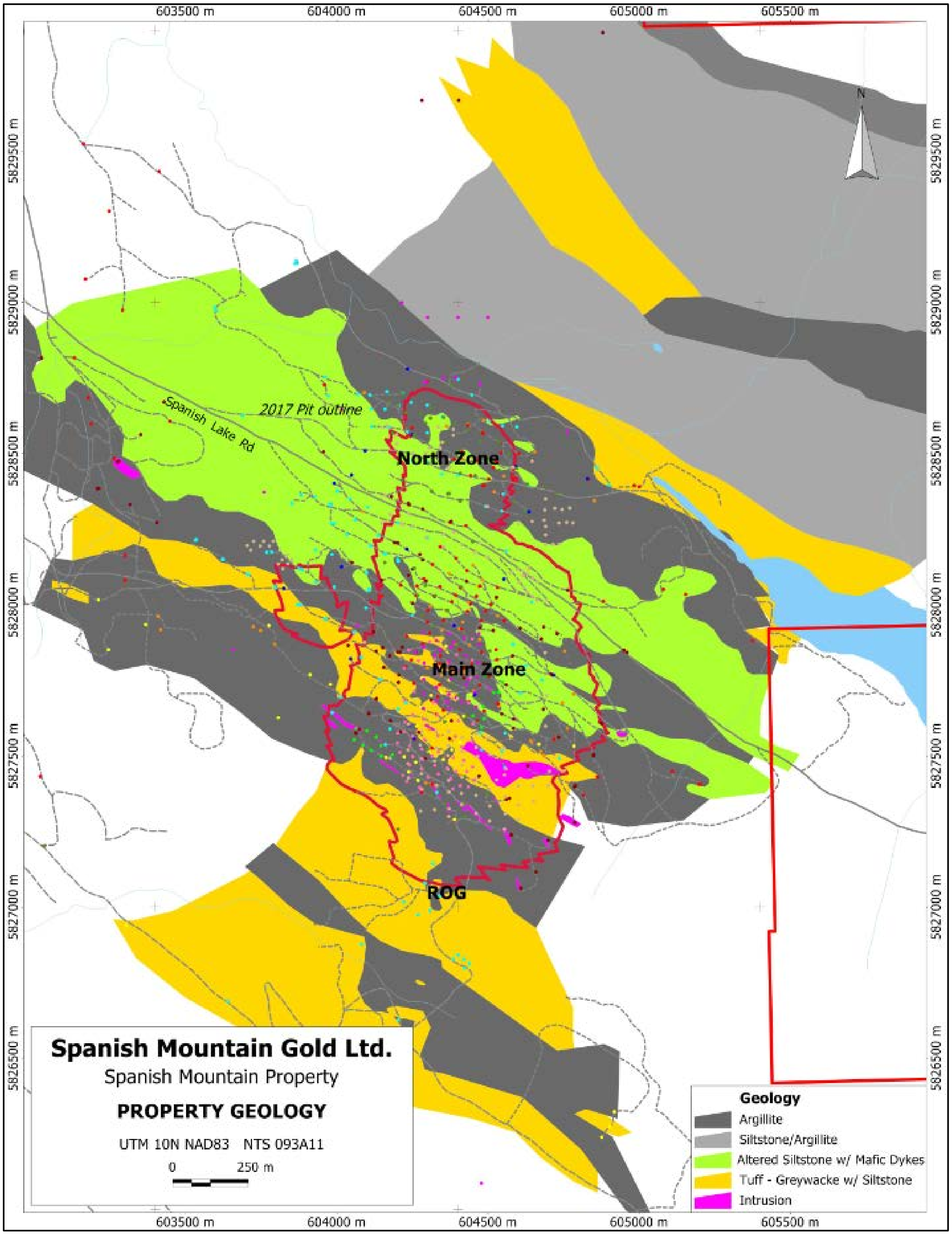


Figure 7-2 Property Geology (approximate pit outline shown in crimson, completely within Claims Boundaries in red)

7.2.1 Stratigraphy

The stratigraphy of the SMG deposit has been summarized by Singh (2008). Slightly revised, it comprises the following stratigraphic sequence from northeast to southwest, and stratigraphically higher to lower:

- **North Zone Argillite**: Fine-grained, black argillite with siltstone interbeds, generally 30 to 100 m thick. Interbeds of altered tuff also occur. This unit hosts wide zones of disseminated gold mineralization. Alteration consists of ankerite, sericite, pyrite, silicification, and quartz veining.
- **Altered (Upper) Siltstone** (with mafic dykes): Medium to light grey, finely laminated, up to 130 m thick. Several altered mafic dykes are present. Visible gold has been noted in quartz veins in several locations. Alteration consists of chromium-rich sericite, ankerite, silicification and quartz veining.
- **Main Zone (Upper) Argillite**: Black, graphitic, locally finely laminated. The unit is up to 100 m thick, with contorted bedding (cataclastic deformation) and is locally friable and faulted. Alteration consists of occasional ankerite and minor quartz veins. The bulk of the disseminated gold mineralization (>65%) is hosted in this unit.
- **Lower Tuff - Greywacke** (with mafic dykes): Often mottled, light to dark grey, fine to coarse-grained tuffs with lesser siltstones, greywackes, and minor felsic dykes. Local argillite horizons are also present. The unit is often strongly silicified, and sometimes pervasive alteration (sericite–ankerite–silica) has made identification of the original rock type very difficult. Visible gold is often found in quartz veins. It also contains thin sills of a probable mafic intrusion.
- **Conglomerate**: Medium–grained, angular to sub-rounded, clast supported. Clasts are commonly siltstone, tuff, and greywacke. The unit is narrow (< 1 m), however, it is useful as a marker horizon at the base of the Lower Tuff – Greywacke sequences.
- **Lower Argillite** (with tuffs and siltstone): Black to dark grey, interbedded argillite, tuff, and siltstone, with minor felsic dykes. This unit exhibits ankerite and silica alteration and only minor graphite. Pyrite content is generally less than 2%. The unit hosts lesser to minor amounts of gold mineralization.

The narrow intrusive felsic sills and dykes, as seen in drill core, have also been noted in outcrop outside the deposit to the southwest, within siltstone-greywacke sequences along the top of the ridge.

Outside of the Main and North Zones, other lithological units have been identified in drill core. These include amygdaloidal basalt to the northeast of the Main Zone in the Placer area, quartz porphyry rhyolite, diorite, and quartz-feldspar porphyry, as seen in drill core in the “Ropes of Gold” (ROG) area, situated south of the Main Zone.

7.2.2 Structure

On a regional scale, the Eureka Thrust has influenced large scale structure on the Property. The Eureka Thrust is a regional scale suture zone marking the western extent of the Omineca Terrane. The trace of the thrust fault lies about 7 km to the northeast of the Main Zone. The major phases of deformation run northwest to north-northwest, parallel to the terrane boundary. The stratigraphic grain of the rocks also runs in a northwest direction.

A compilation of the historical structural data, with a focus on the North Zone, has recently been done by Campbell (2011). Campbell has proposed at least six prominent northwest trending structures at the property scale. He has interpreted these structures as representing either fracture zones or lithological contacts.

Late-stage faulting is indicated by a number of north-easterly to north-north-easterly faults cutting across the Main Zone The most prominent is a fault exposed in an exploration pit, called the Imperial Metals pit, and also intersected in drill core; the fault strikes almost due north. In drill core, numerous graphitic fault zones have been logged. In both surface outcrops and in drill core, there is a lack of continuity on tens of a metre scale, particularly in the North Zone. Gold mineralization has been influenced by this set of late-stage faulting.

Based on recent geological mapping and structural analyses, the geological understanding of the North Zone has increased. It is currently thought that the North Zone argillite is stratigraphically equivalent to the Upper Argillite unit within the Main Zone and is separated by possibly a syncline. This is significant since the majority of the disseminated gold in the Main zone is hosted by the Upper Argillite sequence.

7.2.3 Alteration

The sedimentary package has undergone widespread alteration. The most extensive alteration consists of ankerite-sericite-pyrite, with accessory rutile. Ankerite typically occurs as porphyroblasts up to 10 mm in diameter, which are sometimes stretched parallel to foliation within the black argillite. Within the tuffs/greywackes and intrusive sills, the ankerite is more pervasive, and along with silica alteration, sometimes completely alters the original composition of the rock. Sericite alteration is also locally intense, resulting in a bleached appearance. Silicification has affected the siltstone and tuff units and varies in intensity from weak to strong and pervasive. Bright green chrome mica (fuchsite) occurs as isolated grains within tuffs/greywackes and within intrusive sills, where it also appears as a pervasive green alteration. Ross (2006) identified chrome-bearing spinel in petrographic work within the cores of clots of chrome mica flakes. Both chrome mica and sericite (i.e., mica occurring as a scaly mass) alteration likely occurred at the same time but reflect the different compositions of the rock that was altered.

Pyrite is typically 1 to 2% within the argillite but can be up to 6% locally, and occurs as fine disseminations, as cubes up to 1.5 cm, along veins as blebs, and as fracture fill. Within siltstones, tuffs, and greywackes, it forms larger cubes up to 15 mm, but is generally less abundant. Based on petrographic work by Ross (2006), some of the pyrite may be early diagenetic pyrite, but most appears to be related to quartz-carbonate veins, in variable states of deformation.

7.2.4 Mineralization

Gold mineralization occurs as two main types:

1. Disseminated within the black, graphitic argillite. This is the most economically significant form. Gold grain size is typically less than 30 microns, and is often, but not always, associated with pyrite. Disseminated gold has also been associated with quartz veins within faults zones in the argillite.

2. Within quartz veins in the siltstone/tuff/greywacke sequences. It occurs as free, fine to coarse (visible) gold and can also be associated with sulphides including galena, chalcopyrite, and sphalerite. Highest grades have come from coarse gold within quartz veins.

Disseminated gold within the argillite units is by far the most economically important type of mineralization, and has been traced for over 2 km, occurring in multiple stratigraphic horizons. From drill core, elevated gold content has been noted within fault zones as well as within quartz veins in fault zones. However, the influence of fault zones in relation to the gold content of the deposit is not certain.

Examination of 15 representative core samples of disseminated gold in thin section work by Ross (2006) has concluded the following:

- Native gold (electrum) was identified in four samples, and it occurred as inclusions and fracture fill in pyrite, on crystal boundaries between pyrite crystals and in the gangue adjacent to pyrite. It is very fine grained <20 µm, and generally <5 µm. It is associated with equally fine-grained chalcopyrite-galena-sphalerite, which occurs in all the same habits. All the mineralized samples occurred in variably carbonaceous mudstones/siltstones to fine-grained greywackes, with quartz-carbonate-pyrite veinlets and disseminations. There is no clear indication from this study that the gold is preferentially associated with any habit of pyrite (i.e., disseminated or veinlet, euhedral or subhedral). The deformation state (i.e., degree of cataclastic deformation) of the host rock does not appear to be significant, at least not on the thin section scale; however, a larger scale relationship to position on fold limbs should not be ruled out.

Although a lesser component, quartz veins carrying free gold have yielded the highest gold grade individual samples on the Property. For example, hole 07-DDH-588 intersected 241 g/t Au over 1.5 m in the Main Zone, and hole 11-DDH-950 intersected 106 g/t Au over 0.75 m in the North Zone. These veins tend to occur in the more competent facies such as siltstone and tuff/greywacke. The veins are discontinuous on surface and exhibit a strong nugget effect. The veins have been followed with confidence for about 40 m on the Main Zone. Gold is often associated with base metals in these veins. In particular, sphalerite, galena and chalcopyrite are commonly associated with free gold. Economically, the base metals are insignificant, but mineralogically they are a good indicator of gold mineralization. It is thought that gold and base metals may have been re-mobilized into these veins.

These veins typically crosscut all foliation fabrics and thus appear to have been emplaced late in the tectonic history. From work done by geological mapping and on oriented core data, it is known that the veins generally strike between 010° and 050°, and dip at various angles to the southeast and northwest. Several “blow-out” veins, which are 1 to 5 m thick, have been identified on the Main Zone.

7.3 Comments on Section 7

In the opinion of the QP, William Gilmour, P.Geo., the regional setting and local geology are sufficiently well understood to support the estimation of Mineral Resources.

8 DEPOSIT TYPES

The Spanish Mountain gold deposit is classified as a sediment-hosted vein ("SHV") deposit, as defined by Klipfel (2005). Key characteristics of SHV deposits include the following:

- Hosted in extensive belts of shale and siltstone sedimentary rocks of up to thousands of square kilometres.
- Rocks originally deposited in sequences along the edges of continents known as passive margin settings.
- The sedimentary belts have typically undergone fold/thrust deformation.
- Other important tectonic and structural indicators include proximity to continental basement, the presence of cross structures and multiple episodes of alteration.
- The presence of quartz and quartz-carbonate veins.
- Wide-spread regional carbonate alteration is common; the alteration is typically ankerite, dolomite or siderite, as porphyroblasts and/or as pervasive, fine-grained carbonate.
- Widespread sericitic alteration in both argillite and siltstone.
- Knots and “nests” of pyrite along with large pyrite cubes and fine-grained disseminated pyrite throughout the host rocks and in argillites.
- Are often simple gold systems;
  - sometimes trace elements associated with SHV deposits are arsenic (as arsenopyrite), tungsten, bismuth and tellurium;
  - generally, there is a paucity of copper, lead and zinc sulphides, but minor amounts occur in a few deposits;
- The deposits can be associated with prolific placer gold fields.
- Granitic rocks commonly, but not always, occur in spatial association with the deposit. The timing of granitic intrusion can be before or after mineralization.

SHV deposits are some of the largest in the world with many of the largest located in Asia, especially in Russia. Examples include Muruntau (>80 million (“M”) ounces ("oz")); Sukhoy Log (>20 M oz); Amantaytau, Olympiada (both >5 M oz) and others. In Australia they include Bendigo (>20 M oz), Ballarat, Fosterville, and Stawell. In North America, small to medium size deposits occur in the Meguma Terrane of Nova Scotia and in the southern half of the Seward Peninsula in Alaska (Klipfel, 2005).

The SMG deposit shows many of the features common to these deposits (Klipfel, 2007), including some of the structural characteristics, regional extent of alteration, alteration mineralogy, mineralization style and gold grade. In addition, the metal chemistry is gold without an association of other trace elements. There is also a lack of significant base metal sulphides.

8.1 Comments of Section 8

In the opinion of the QP, William Gilmour, P.Geo., the SHV gold model is appropriate to use for exploration vectoring.

9 EXPLORATION

This PFS Report is concerned primarily with Resources in the Main and North Zones, which are based on the results of sampling of both drill core and RC cuttings from the programs carried out from 2005 to 2018. Thus, a summary is provided of the work done in these programs. Programs carried out before 2005 are summarized in Section 6 – Exploration History. The 2005 to 2009 exploration programs carried out by SMG were done under its former name of Skygold Ventures Ltd. A more complete summary of the drilling programs from 2004 to 2020 is contained in Section 10.

The exploration described in this Section is a summary of geological, geochemical, and geophysical programs (Table 9-1).

Table 9-1 Summary of SMG Exploration and Drilling Programs

Year	Work Done
2020	527 m of NQ drilling in 6 holes and 4,490 m of RC drilling in 28 holes. Geotechnical program
2018	1,061 m of HQ drilling in 6 holes and 1,091 m of RC drilling in 11 holes. Extensive archaeological impact assessment
2014	2,621 m of RC drilling in 18 holes
2013	9,226 m of RC drilling in 56 holes
2012	27,310 m of core drilling in 144 holes plus 12 geotechnical holes. 2,012 m of core drilling of North Cedar Zone.
2011	19,437 m of core drilling in 82 holes; for exploration and geotechnical purposes. 32 exploration core holes in the North Cedar Zone. Soil sampling. Airborne geophysical survey. Baseline environmental studies.
2010	6,834 m of core drilling in 20 holes; for exploration, geotechnical and metallurgical purposes. Baseline environmental studies.
2009	13,769 m of core drilling in 62 holes. Geological mapping, rock and soil sampling.
2008	40,449 m of core drilling in 161 holes. Geological mapping, rock and soil sampling.
2007	29,993 m of core drilling in 126 holes. Geological mapping, rock and soil sampling. Metallurgical test work on drill core.
2006	21,886 m of core drilling in 88 holes. 5,008 m of RC drilling in 50 holes. Geological mapping, rock and soil sampling. Airborne geophysics and ortho-photography on a property-wide scale. Environmental baseline studies.
2005	7,746 m of core drilling in 35 holes. 3,377 m of RC drilling in 30 holes. Geological mapping, rock and soil sampling.

9.1 2005 Program

Aside from drilling (see Section 10) a program of geological mapping, rock sampling and soil sampling was carried out in 2005. Much of the geological mapping and rock sampling was carried out over the Don claims, currently known as the North Zone.

Rock samples were analyzed for metallic gold and for 28 elements by ICP methods. A limited number of analytical standards were submitted with the rock samples. The results indicated no significant variation in gold analysis. No record of the number of samples or analytical results is available.

Soil samples were collected on the March 1 claim, west of the Don claims. They were analysed by aqua regia /AA methods for gold and for 28-elements by ICP. No record of the number of samples, sample spacing, or analytical results is available.

9.2 2006 Program

Grid soil sampling (1,515 samples) was completed in 2006. Most samples were collected on an approximately 50 m by 100 m grid, totalling 36-line km. Values up to 865 ppb gold led to the discovery of the Oscar showing, north of Spanish Creek. No complete record of analytical results is available.

Rock samples, totalling 465, were collected on a regional scale. Values up to 2.1 g/t gold also assisted in the discovery of the Oscar showing. No complete record of analytical results is available.

Surface samples were collected using standard practices and techniques by experienced geologists and/or well supervised technicians.

Geological mapping and prospecting continued across the Main Zone at 1:100 scale and regionally at 1:5,000. This mapping proved effective for correlating geophysical relationships across the Property.

Geophysical work comprised an airborne electromagnetic and magnetic survey over the Property by Fugro Geosciences. The data showed a positive correlation between more resistive wacke sequences and more conductive argillite sequences over the Main Zone. By inference, the contact between the wacke and argillite, which is integral to localizing gold mineralization in the Main Zone, was projected across the geochemical soil anomaly.

Other airborne work included orthophotographs, which were produced on a 1:1000 scale with a 0.3 m resolution and topographic maps, were produced with precise 2-m contours.

In addition, Knight Piésold Ltd. was contracted to perform environmental baseline studies, which included meteorological studies, surface water hydrology and quality studies, preliminary waste rock characterization and fisheries sampling.

9.3 2007 Program

A program of limited geological mapping and prospecting, and the collection of 182 rock samples were completed. At least 28 of these samples contained >5 g/t gold. The samples were collected mainly in quartz veins with base metal mineralization. The best results were obtained from greywacke sequences south of the Main Zone. No complete record of analytical results is available.

A total of 792 soil samples was collected at 25 m intervals on lines with 100 m or more spacing, along 13 km of grid lines. This sampling extended and infilled pre-existing soil grids. Samples were analysed by multi-element ICP methods. No complete record of analytical results is available.

Metallurgical testing involved the analysis of four composite samples by various flotation techniques to determine preliminary gold recoveries. The testing determined the following:

- The gold is relatively fine and occurs frequently as binary with pyrite,
- The gold is easily recovered by flotation,
- The sulphide concentrate responds very well to carbon-in-leach cyanidation, with gold recovery of about 95%.

In addition, a 30-person camp and core logging facility were built on SMG's private property located within the village of Likely.

9.4 2008 Program

Geological mapping and rock sampling were mainly carried out on outcrops, and fault zones in the Main Zone, which had been recently exposed by pad building. Mapping was also done in the ROG and Cedar Creek areas. About 35 rock samples were collected, with about 6 samples containing >5 g/t gold. No complete record of analytical results is available.

In total, 341 soil samples were collected between the Main Zone and the ROG area to the south. These results further outlined a northwest trend that corresponds with the strike of the Main Zone stratigraphy. No complete record of analytical results is available.

Environmental baseline studies were limited to monitoring weather stations.

9.5 2009 Program

Reconnaissance geological mapping, rock sampling (41 rock samples) to the north on Black Bear Mountain, and preliminary re-interpretation of historic data were carried out. Geological mapping and related work resulted in the recognition of a northeast-trending steep structure believed to control mineralization. Fe-Mg-carbonate alteration forms a 5 km to 8 km halo around the Main Zone resource.

The Imperial Metals pit and neighbouring trenches on the Main Zone were re-excavated, mapped and chip sampled.

A limited soil sampling program was carried out in the ROG area (121 samples) and the Cedar Creek - Mt Warren area (28 samples). No complete record of analytical results is available.

9.6 2010 Program

Baseline environmental studies conducted by Knight Piésold Ltd. continued in 2010 as part of a long-term data collection and monitoring program. The 2010 work included meteorology, surface hydrology, stream water quality analysis, and flora and fauna studies. The size of the Property was increased with the acquisition of the Cedar Creek property to the west.

9.7 2011 Program

Exploration work was also performed in the southeast part of the Property. A grid soil survey was performed, outlining a copper anomaly.

Other work included an airborne geophysical survey, which was carried out over the Property in late 2011, as well as an airborne LiDAR topographic survey.

The magnetic and DIGHEM V® electromagnetic airborne survey was carried out by Fugro Airborne Surveys Ltd. Results of the work showed that around the East Block, the sedimentary units exhibit low magnetic susceptibility, with a range of only about 200 nT. Within the non-magnetic units, however, there are linear lows and highs that have been attributed, respectively, to faults or weakly magnetic intrusions. The local geological strike is roughly 130° (±10°), but the strikes of the inferred faults and dykes are quite variable. The southeast-trending geology is intersected by a weakly magnetic dyke-like linear feature that extends south-southwest.

A broad area of low resistivity, defined as Zone A, occurs throughout most of the Eastern Block. Numerous discrete electromagnetic conductor anomalies have been identified within Zone A. The highly conductive and non-magnetic characteristics suggest graphite as a probable causative source.

For the topographic survey, airborne LiDAR technology was used to measure elevations, producing a digital file with high resolution contours at 1-metre scale over the surveyed area. In addition, an orthophoto was produced which was subsequently colour balanced, made seamless and rectified to a resolution of 30 cm. The orthophoto, which is basically an aerial photograph, has also been corrected for topographic relief, camera tilt and lens distortion. The resulting detailed surficial topography outlines forest cover, roads, and geomorphological features. This survey aided in the further exploration of the Property.

Baseline environmental studies continued through the year.

9.8 2012 Program

SMG continued definition drilling with an infill core drilling program on the Main and North Zones.

9.9 2013 Program

SMG conducted an RC drilling program, which focussed on a test block within the deposit on the Main Zone.

9.10 2014 Program

Additional RC drilling was carried out on the Main and North Zones.

9.11 2018 Program

The first stage of drilling comprised three metallurgical HQ holes, totalling 512 m, on the Main Zone. The vertical holes were drilled for confirmatory metallurgical testwork.

The second phase comprised three exploratory HQ holes, totalling 549 m, on the Phoenix Zone which were drilled to test the continuity of mineralization along a one-kilometre-wide corridor outlined by previous work.

SMG also carried out an 11-hole RC drilling program on the Main Zone.

An extensive archaeological impact assessment was done throughout the Property in 2018. Terra Archaeology Ltd explored the area outside of the Main zone, as that area has a previously completed archaeological impact assessment. One archaeological site was identified, where one piece of lithic debitage was collected. The piece consists of fine-grained volcanic chip debitage. The remainder of the areas examined had negative test results, and thus low archaeological potential.

9.12 2020 Program

SMG conducted a 28-hole, 4,490 m RC drilling program, focusing on the Main and North Zones of the deposit.

In addition, a 6-hole NQ core drilling program was carried out in the south-east part of the Property, in the area that is proposed as a Tailings Storage Facility. The condemnation drilling program was designed to test the gold potential of the lithologies underlying this area.

A geotechnical program was also conducted, which involved the completion of 26 geotechnical drill holes and 89 test pits. This work was monitored by Knight Piésold Ltd.

In Bill Gilmour’s opinion, the exploration programs completed to date are appropriate for the style of the mineralization and prospects located on the Project. There are a number of targets prospective for further exploration on the property.

10 DRILLING

In 2004, Wildrose Resources Ltd. carried out drilling on the Property. SMG has been drilling on the Property since 2005. Table 10-1 summarizes the drilling activity on the deposit from 2004 to 2020.

Table 10-1 Summary of Exploration Drilling Programs

Year	Drill Type	No. of Holes	Metres	Core size
2020	RC	28	4,490	n/a
2020	core	6	527	NQ
2018	RC	11	1,091	n/a
2018	core	3	549	HQ
2014	RC	18	2,621	n/a
2013	RC	56	9,226	n/a
2012	core	131	24,290	NQ
2011	core	82	19,437	NQ / HQ3
2010	core	20	6,834	NQ / HQ / HQ3
2009	core	62	13,769	NQ / HQ
2008	core	161	40,449	NQ / NQ2
2007	core	126	29,993	NQ
2006	core	88	21,886	NQ
2006	RC	50	5,008	n/a
2005	core	35	7,746	NQ
2005	RC	30	3,377	n/a
2004	RC	34	2,506	n/a

For the 2005, 2006, and 2007 core drill programs, drilling was contracted to LDS Diamond Drilling of Kamloops BC. The 2008 core drill program was contracted to North Star Drilling. In 2007, the main drill direction of 210 degrees was less than optimal, as it is sub-parallel to known regional faults. Drill directions were changed slightly, and consequently, this reduced the number of holes lost in fault zones.

The 2004 RC drill program was carried out by Northspan Exploration of Kelowna, BC. The 2005 and 2006 RC drilling were done by Drift Exploration Drilling of Alberta, and Northspan Exploration of Kelowna BC.

For the 2010, 2011, 2012, and 2018 programs, core drilling was contracted to Atlas Drilling Company of Kamloops, BC. Downhole measurements including azimuth and dip, were measured using a Reflex EZ-Shot® tool and were collected every 50 m down hole. Collar locations were initially surveyed using a hand-held GPS. Once drilling was completed, the 2010 drill collar locations were more accurately surveyed by

Crowfoot Surveys of Kamloops, BC, utilizing standard surveying equipment. Surveying in 2011 and 2012 was done in-house using Trimble R8R2K Survey® GPS equipment supplied by Cansel Survey Equipment Inc.

For the 2013, 2014, 2018, and 2020 programs, RC drilling was contracted to Northspan Explorations Ltd, of Kelowna, BC. Drilling was done using a skid-mounted Super Hornet drill utilizing five-foot drill rods. A 5.5-inch (140 mm) casing was run through the overburden into solid bedrock, followed by a 4.0-inch (102 mm) diameter drill bit for sample collection. A couple of holes were drilled with a 3.5-inch diameter bit. All samples below the casing represented five-foot (i.e., 1.52 m) sections of rock cuttings, equivalent to rod length.

The RC drill uses a carbide-tipped drill bit attached to a downhole hammer and is powered by compressed air. Rock cuttings, consisting of rock chips of variable size fractions (from about 2 cm size chips to dust size particles) generated by the hammer, travel up the centre chamber of the rods to the surface along with the forced air, where they pass into a cyclone separator.

There was no downhole surveying of RC drill holes. For core drilling, any downhole surveying is mentioned below where applicable. Drill recovery is general good, although RC recovery appears to be better than core recovery.

The locations of the 2004 to 2020 drillholes are shown subsequently in Figure 10-1 through Figure 10-14. Representative examples of drill sections through the mineral deposit are shown in Figure 10-15 and Figure 10-16.

The following is a summary by year of the drilling programs carried out on the Property.

10.1 2004 Program

In October and November of 2004, a Reverse Circulation drilling (RC drilling) program was conducted, to follow up on trench, soil, and geophysical results. In total, 2,504 m were drilled in 34 holes, within areas of known mineralization on the CPW claim and in areas up to 1.3 km away. Drilling was carried out by Northspan Exploration of Kelowna, BC. The RC drilling was supervised by Robert Johnston, P.Geo. Analytical work was performed by Acme Laboratories of Vancouver, BC, an independent assay laboratory.

Some 55 intersections greater than 1 g/t Au were obtained. The most important result was the discovery of a northern extension to the LE (Imperial) Pit area currently known as the Main Zone. Three holes returned long intersections of consistent 1 to 2 g/t Au in unaltered argillite and siltstone.

New mineralization was encountered 700 m and 1000 m west, in the McKeown Placer area of the LE Pit.

10.2 2005 Program

In 2005, SMG began core drilling and continued with RC drilling with joint venture partner Wildrose. The programs comprised 7,746 m of core drilling (35 holes) and 3,377 m of RC drilling (34 holes) in the Main Zone and to a lesser extent in the North Zone.

The RC drilling (140 mm diameter) was contacted to Drift Exploration Drilling of Alberta. During May to June 1,677 m (16 holes) of drilling were completed on the Main Zone and to a lesser extent in the North

Zone. The RC drilling was supervised by Robert Johnston, P.Geo. Analytical work was performed by Eco-Tech Laboratories of Kamloops, BC, an independent laboratory.

This program was designed to follow up 2004 RC drilling. The most significant result was 56 m of 1.17 g/t Au, 330 m north of the LE pit in the Main Zone.

A second phase of RC drilling was carried out during September to November. Nine holes totalling 1,160 m were drilled by Northspan Exploration of Kelowna, BC, on the Don claims, currently known as the North Zone. RC drilling comprised 14 holes totalling 1,700 m, in the North Main zone and reconnaissance drilling testing geochemical anomalies.

The RC drilling programs located a 330 m long zone of consistent 1 to 2 g/t gold mineralization in the area north of the LE Pit. The deepest hole was to 174 m. One hole returned 1.43 g/t Au over 107 m. A second hole returned 39 m of 1.04 g/t Au. Significant mineralization occurred in graphitic shear zones.

From July to November, core drilling was carried out in the LE Pit area within the Main Zone. Diamond drilling had not been utilized since 1988 due to poor core recoveries at the time and the associated possible loss of gold values. Core recoveries were generally greater than 95% except in areas of intense faulting where recoveries were lower and variable. In total, 35 holes totalling 7,746 m were drilled. LDS Diamond Drilling of Kamloops, BC, carried out the drilling.

Analytical work was performed by Eco-Tech in Kamloops. The core drilling was supervised by Robert Johnston, P.Geo.

Two holes totalling 146 m were drilled on the Northern CPW claim, currently known as the Main Zone. Both holes encountered problems with faults and were abandoned short of their target depths. However, encouraging results came from the top 30 m in one of the holes, where 29 m of 0.48 g/t Au was encountered.

Three holes totalling 390 m were drilled on the Western Placer Area and returned intervals of anomalous mineralization, including 153 m of 0.24 g/t Au.

Samples were collected for bulk specific gravity determinations.

10.3 2006 Program

In 2006, SMG expanded its exploration work by NQ core drilling 21,886 m in 88 holes on both the Main and North Zones. This drill campaign led to the discovery of continuous mineralization along a 1.2 km north-south corridor. Several holes intersected significant widths of mineralization between 10 m and 130 m of greater than 1.0 g/t Au.

In addition, 5,008 m of RC drilling in 50 holes were drilled along the eastern edge of the Main Zone, the North Zone, the Placer area west of the Main Zone, and the Cedar Creek area.

The RC program was performed again by Northspan Explorations of Kelowna, BC. A comprehensive system of QA/QC was conducted, involving rigorous sample collection and handling procedures, as done previously in 2005.

The holes were drilled at a -60° dip, expect for areas of expected deep overburden.

The results of the RC drill program were that significant gold anomalies were encountered in 16 of the 50 holes drilled. A large area of anomalous gold values was found on the western side of the Property. The best of these holes included 26 m of 0.30 g/t Au and 55 m of 0.26 g/t Au. No significant intersections were encountered in the Cedar Creek drill holes.

Seven holes were drilled in the North Zone. Significant intersections included 11 m of 0.32 g/ t Au, and 7.6 m of 0.47 g/t Au.

The Placer area west of the Main Zone also had significant mineralization, with 9 of the 21 holes returning gold values. Best intersections were 26 m of 0.32 g/t Au; and 26 m of 0.30 g/t Au.

Samples were collected for bulk specific gravity determinations.

10.4 2007 Program

In 2007, SMG conducted 26,993 m of NQ core drilling in 126 holes, focussing on infill drilling on the Main Zone for geological resource modelling, but also testing outlying areas. Overall, the infill drilling expanded the grade and width in the Main Zone. It successfully tested for stratigraphic continuity in the Main Zone between the Main Zone argillite and the Lower wacke sequence. In addition, Main Zone stratigraphy was extended as far as 500 m to the northwest, approaching but not including the Placer area. One hole in this area intersected 37 m of 1.01 g/t Au in the argillite unit of the Main Zone. In the south part of the Main Zone, high grade mineralization was intersected, with one hole having 42 m of 9.38 g/t Au, in the wacke unit.

An EZ-Mark downhole core-orientation tool was used for selected drill holes. Orientated data were collected for bedding measurements, quartz vein measurements and fault zone measurements. Analysis for these data sets corresponds with surface structural data.

No drilling was done in the North Zone. Re-interpretation of the data suggested that mineralization in this area may be steeply dipping. It also suggested that the North Zone structurally overlies the Main Zone.

10.5 2008 Program

A large drilling program consisting of 40,449 m of NQ and NQ2 core drilling in 161 holes was completed in 2008. Drilling focussed on the lateral extent of the Main Zone, to the northwest and to the north at depth, and the lateral extent of the North Zone, for a total of 140 holes. Drilling also tested the ROG area, where high-grade trench and rock samples were targeted with 18 drill holes; the Cedar Creek area, where two drill holes tested anomalous gold in soils; and the Placer area where one drillhole tested an area of an anomalous rock sample.

An EZ-Mark downhole core-orientation tool was used for selected drillholes. Orientated data were collected for bedding measurements, quartz vein measurements and fault zone measurements. Analysis for these data sets corresponds with surface structural data.

10.6 2009 Program

In 2009, definition drilling continued in the Main Zone with a program of 62 core drill holes, totalling 13,769 m. Of these holes, 33 HQ holes were drilled in the Main Zone, along with four twinned NQ holes, to test whether there was any apparent bias in analytical gold grades in NQ versus HQ size core. The results were inconclusive, since the HQ samples were analysed without the insertion of standards and at a different lab from the NQ samples.

To test for mineralization below the Main Zone resource, an addition three deep holes were drilled below the Main Zone, ranging in depth from 450 m to 650 m, and totalling 1,705 m. The holes were collared about 200 m apart along a fence oriented from 119° to 289°. The drill holes intersected thick sequences of sedimentary strata with generally low gold values at depth. Major faults encountered in drilling may represent feeder structures to known mineralization.

Other drill targets were also core drilled, including the ROG, Cedar Creek, Placer, North Zone step-out and Black Bear Mountain, for a total of 6,849 m in 21 holes.

10.7 2010 Program

In 2010, drilling comprised 20 core drill holes within and peripheral to the Main and North Zones of the deposit, for a total of 6,834 m. Seven of the holes were geotechnical holes of HQ3 size within the Main and North Zones. The sites targeted areas of potential waste rock, which will possibly form the pit walls. Four metallurgical (HQ) holes were drilled in the Main and North Zones. These holes were designed to provide information for the on-going metallurgical testing program dealing with gold recoveries. One HQ3 hole, located in the Main Zone, was selected for both geotechnical and metallurgical analysis. The remaining eight NQ holes were exploration holes drilled outside the boundary of the Main and North Zones, to determine the potential for expansion of the Main/North Zone gold resource.

Drilling was contracted to Atlas Drilling Company of Kamloops, BC. Downhole measurements including azimuth and dip were measured using a Reflex EZ-Shot® tool. The measurements were collected every 50 m down hole. Drill collar locations were surveyed in UTM Zone 10N, using NAD83 Datum. Survey work was completed by Crowfoot Surveys of Kamloops BC, utilizing standard surveying equipment.

The western edge of the Main Zone was explored by three holes, and all three encountered gold mineralization. A high-grade zone was intersected in one hole, resulting in 39 m of 0.43 g/t Au.

The North Zone was explored by 3 holes, with intercepts including 76 m grading 0.87 g/t Au. These holes extended the known mineralization in the North Zone by 100 m to the southwest and to the southeast by 110 m.

Long intersections of gold mineralization were outlined in the Main Zone from several of the geotechnical holes. The longest section contained 121 m of 0.74 g/t Au. The second hole ran 1.15 g/t Au over 42 m.

10.8 2011 Program

In 2011, SMG carried out an infill drilling program on the Main and North Zones, for a total of 82 holes. This work totalled 8,869 m of core drilling from 31 holes in the Main Zone, and 10,568 m of core drilling from 51 holes in the North Zone. The program was designed to provide additional information to enable a re-classification of resources from the Inferred to the Measured and Indicated categories. Included in the Main Zone were three deep holes drilled to test for mineralization at depth. These holes reached depths of 444 m, 566 m, and 517 m, respectively. One of the holes encountered 23 m of 0.58 g/t Au at a depth of 484 m; a second hole carried 9.0 m of 1.32 g/t Au at a depth of 489 m, indicating that gold mineralization continued to depth. In addition, four of the holes were geotechnical holes, designed to provide information for open pit designs.

A core drilling program was undertaken in the North Cedar area where 32 core drill holes were drilled in a grid-like pattern at intervals of roughly 500 m. Within this area, a new zone of gold mineralization was discovered in late 2011 and termed the Phoenix Zone. This zone is located about two km west of the Main Zone. Gold intercepts included 92 m grading 0.58 g/t Au, and 55 m grading 0.82 g/t Au.

On the southeast part of the Property, near the upper parts of Cedar Creek, a drill program, consisting of 17 core drill holes, resulted in low concentrations of copper over wide intervals, with narrow intervals having higher values over the range of 0.11 to 0.44% copper.

10.9 2012 Program

In 2012, SMG continued definition drilling with an infill core drilling program on the Main and North Zones, which comprised 144 core drillholes for a total of 27,310 m. Work focussed on 131 NQ core drill holes, for a total of 24,290 m to determine the potential for expansion of the Main/North gold resource. This work totalled 19,970 m of core drilling from 98 holes in the Main Zone, and 4,320 m of core drilling from 33 holes in the North Zone and was used for an updated 2012 Resource Estimate. In addition, 12 geotechnical (HQ) drill holes on the Main and North Zones provided information on rock competencies, to aid in the design of a potential open pit.

Exploration drilling continued in the North Cedar area to better define the Phoenix Zone, resulting in seven core drill holes totalling 2,012m.

The work confirmed the style of the gold mineralization as both disseminated gold within argillite and argillite-siltstone horizons, and as gold in quartz veins within these units. Altered tuffs are also shown to contain significant gold mineralization.

10.10 2013 Program

In 2013, a review by Dr. Morris Beattie, P.Eng. then CEO of SMG, compared gold grade determinations of core drilling (2005 to 2012) versus RC drilling (2004 to 2005). Based on this review, it was concluded that the sample size provided by the sub-sampling of the NQ drill core resulted in an understated grade for the deposit. A limited comparison of grades from selected core drillholes and nearby (<7 m) RC holes suggested that a negative bias occurred in the sampling from the core drilling.

The Beattie 2013 report concluded that larger sample sizes produced by RC drilling are expected to give a more accurate gold grade, since the larger volume of rock gives more representative samples of gold grains than does split, half-core samples. Furthermore, gold grades are also expected to be more accurate due to significantly better recovery in gouge and fault zones.

Based on the conclusions of this study, SMG conducted an RC drilling program, which focussed on a test block within the deposit on the Main Zone. In total, 9,226 m were drilled in 56 RC holes.

For the 2013 program, RC drilling was contracted to Northspan Explorations Ltd, of Kelowna, BC. Drilling was done using a skid-mounted Super Hornet drill utilizing five-foot drill rods. A 5.5-inch (140 mm) casing was run through the overburden into solid bedrock, followed by a 4.0-inch (102 mm) diameter drill bit for sample collection. A couple of holes were drilled with a 3.5-inch diameter bit. All samples below the casing represented five-foot (1.52 m) sections of rock cuttings, equivalent to a rod length.

10.11 2014 Program

Additional RC drilling was carried out on the Main and North Zones in 2014, totalling 2,621 m in 18 holes.

10.12 2018 Program

In 2018, SMG carried out a two-stage core drilling program. Drilling was contracted to Atlas Drilling Company of Kamloops, BC. Downhole measurements including azimuth and dip were made using a Reflex EZ-Shot® tool. The measurements were collected every 50 m down hole. Drill collar locations were surveyed in-house using Trimble R8R2K Survey GPS equipment.

The first stage of the 2018 program comprised three metallurgical HQ holes (as these were not exploration drilling, they are not included in Table 10-1), totalling 512 m, on the Main Zone. The vertical holes were drilled for confirmatory metallurgical testwork. This work provided detailed information required for the design and costing of any future process plant.

The complete core was transported to SMG's core logging facility, where rock quality designation (RQD) procedures and core logging were completed. The core was then stored on pallets and transported by a trucking company to an independent metallurgical laboratory in Reno, Nevada.

Logging of the core showed that the three holes encountered sequences of argillite ± siltstone, followed by a sequence of greywacke. The argillite sequences contained quartz veins that are typically pyrite rich and contain occasional galena, chalcopyrite, and sphalerite.

The second phase of the 2018 program comprised three exploratory HQ holes, totalling 549 m, on the Phoenix Zone, which were drilled to test the continuity of mineralization along a one-kilometre-wide corridor outlined by previous work. This program was in an area separate from the Mineral Resource area.

In September 2018, SMG continued exploration with an infill RC drilling program. The work was done on the Main Zone, with the goal of bringing inferred resources up to the measured and indicated categories. Drill sites were selected by Moose Mountain Technical Services, of Cranbrook, BC. The program comprised 11 RC drillholes for a total of 1,091 m.

Drill collar locations were surveyed in-house using Trimble R8RTK Survey GPS equipment supplied by Cansel Survey Equipment Inc.

10.13 2020 Program

In 2020, SMG carried out a core drill hole program**.** Drilling was contracted to Atlas Drilling Company of Kamloops, BC. The location of the holes was determined by SMG. After completion, the holes were cemented.

The drilling was carried out in the area of the proposed Tailings Storage Facility, as condemnation drill holes to evaluate the underlying geology for gold potential prior to this area being reserved as a possible tailings storage site.

Six holes (20-CCR-043 to 048), totalling 527 m, were drilled, one to 42 m depth and five to 97 m depth. The complete core was transported to SMG's core logging facility, where rock quality designation (RQD) procedures and core logging were completed. Analytical results indicate that the drill holes contained no significant gold values.

In 2020, a program of RC drilling was completed with 4,490 m drilled in 28 holes (20-SMRC-1231 to 1258). Drilling took place in both the Main and North Zones. Drill sites were selected by Moose Mountain Technical Services, of Cranbrook, BC. Drill collars were surveyed in-house using Trimble R8RTK Survey GPS equipment supplied by Cansel Survey Equipment Inc.

As of the end of December 2020, all the 2020 RC samples had been received by the ALS laboratory in Kamloops. However, as of the Effective Date of the Mineral Resource (Section 14), which was February 3, 2021, SMG had not received any results from ALS. Therefore, any analytical results for the 2020 RC drilling are beyond the scope of this Report, pertaining to the Mineral Resource, and therefore there were no results to include in the Mineral Resource calculations.

10.14 Comments on Section 10

In the opinion of the QP, William Gilmour, P.Geo., the quantity and quality of the data collected in the completed drill hole programs are sufficient to support the Mineral Resource Estimation. There are no known drilling, sampling or recovery factors that could materially impact the accuracy and reliability of the results.

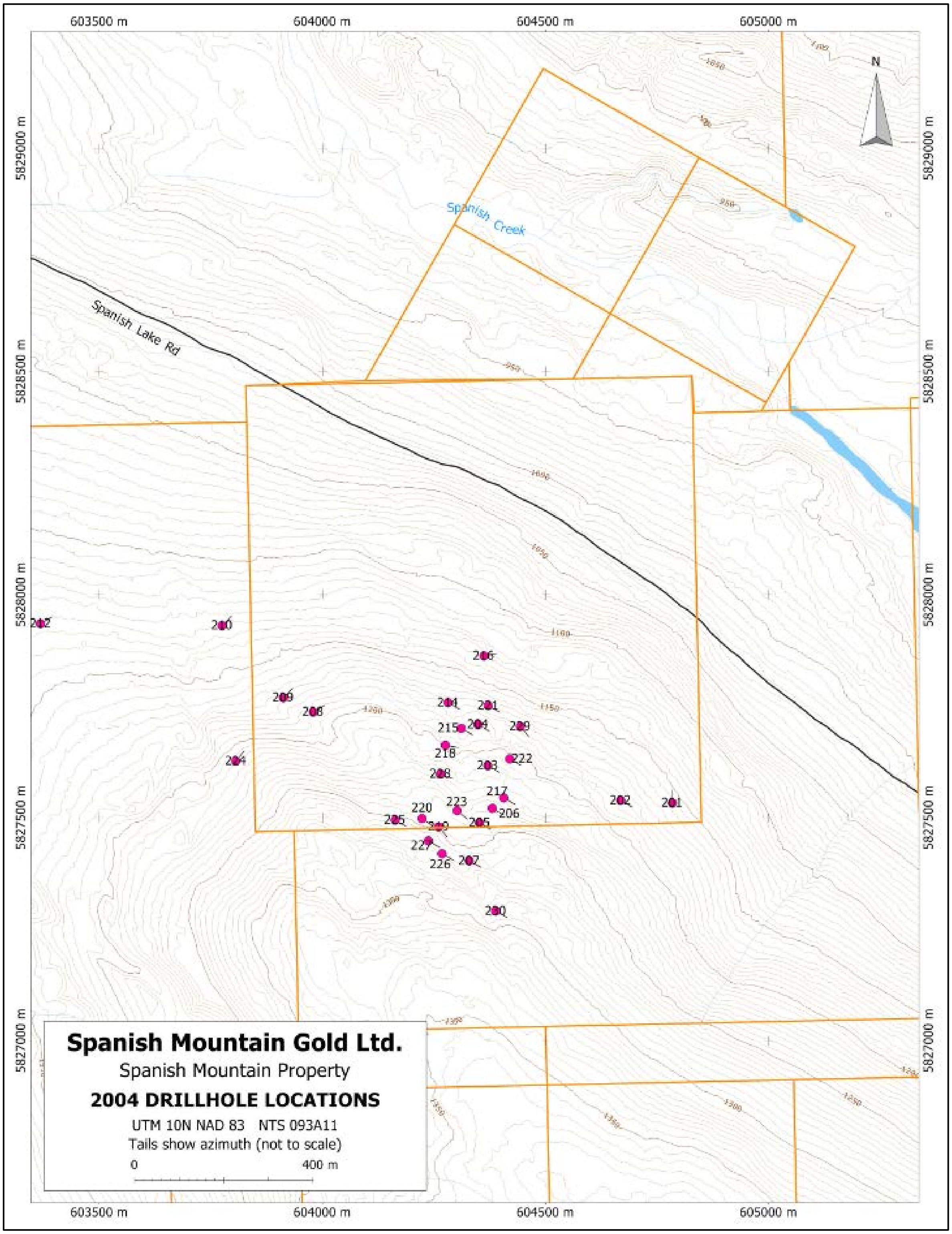


Figure 10-1 2004 Drillhole Locations

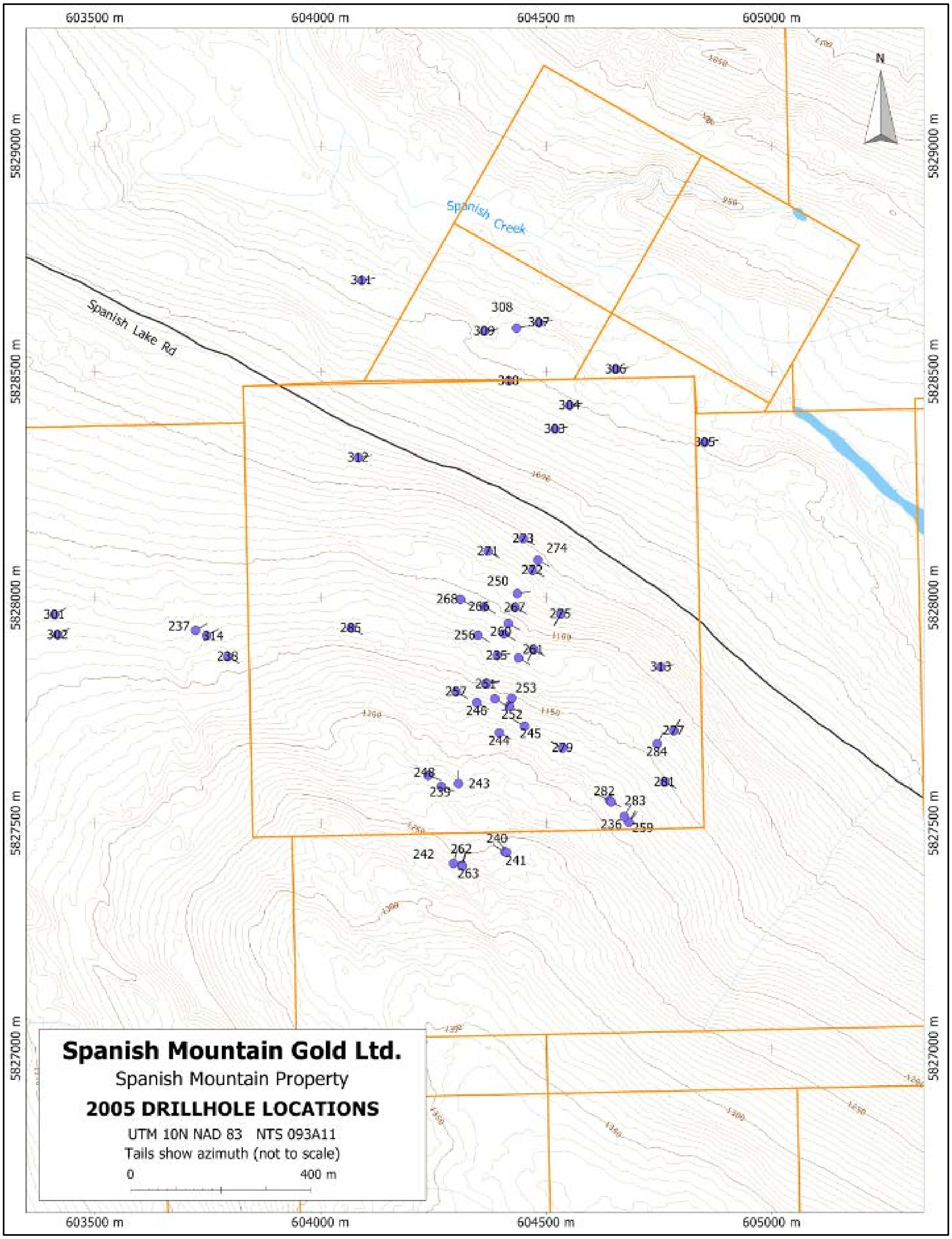


Figure 10-2 2005 Drillhole Locations

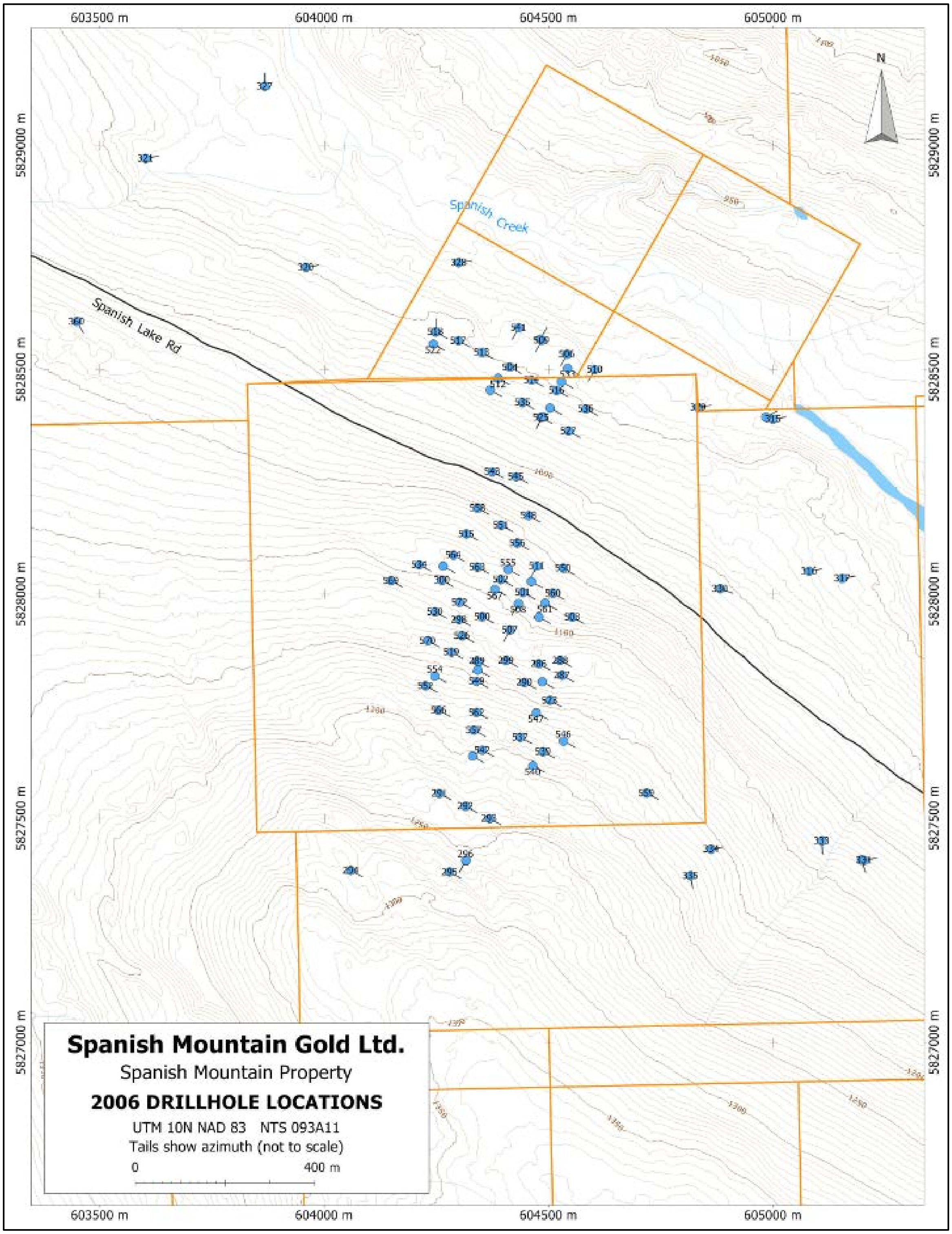


Figure 10-3 2006 Drillhole Locations

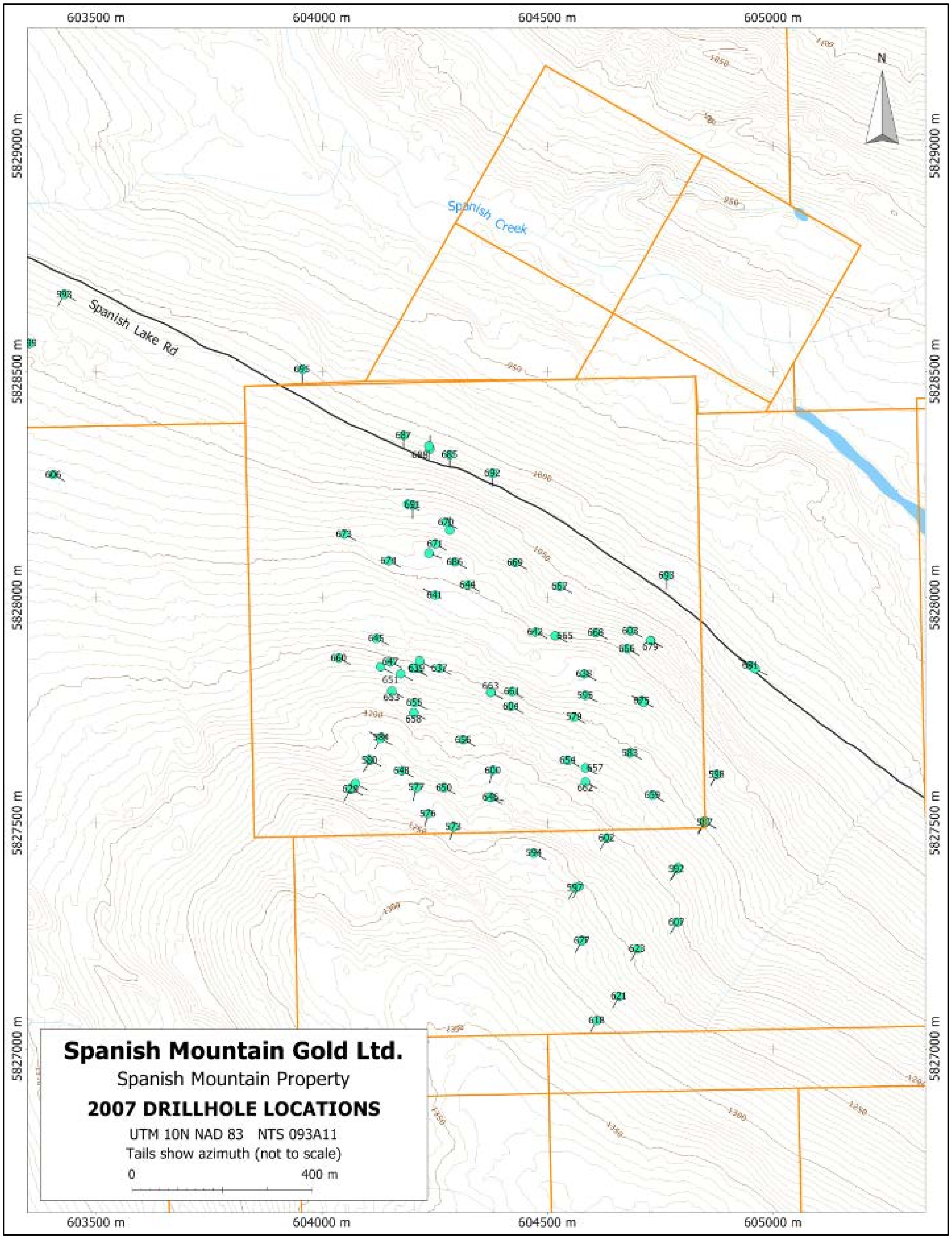


Figure 10-4 2007 Drillhole Locations

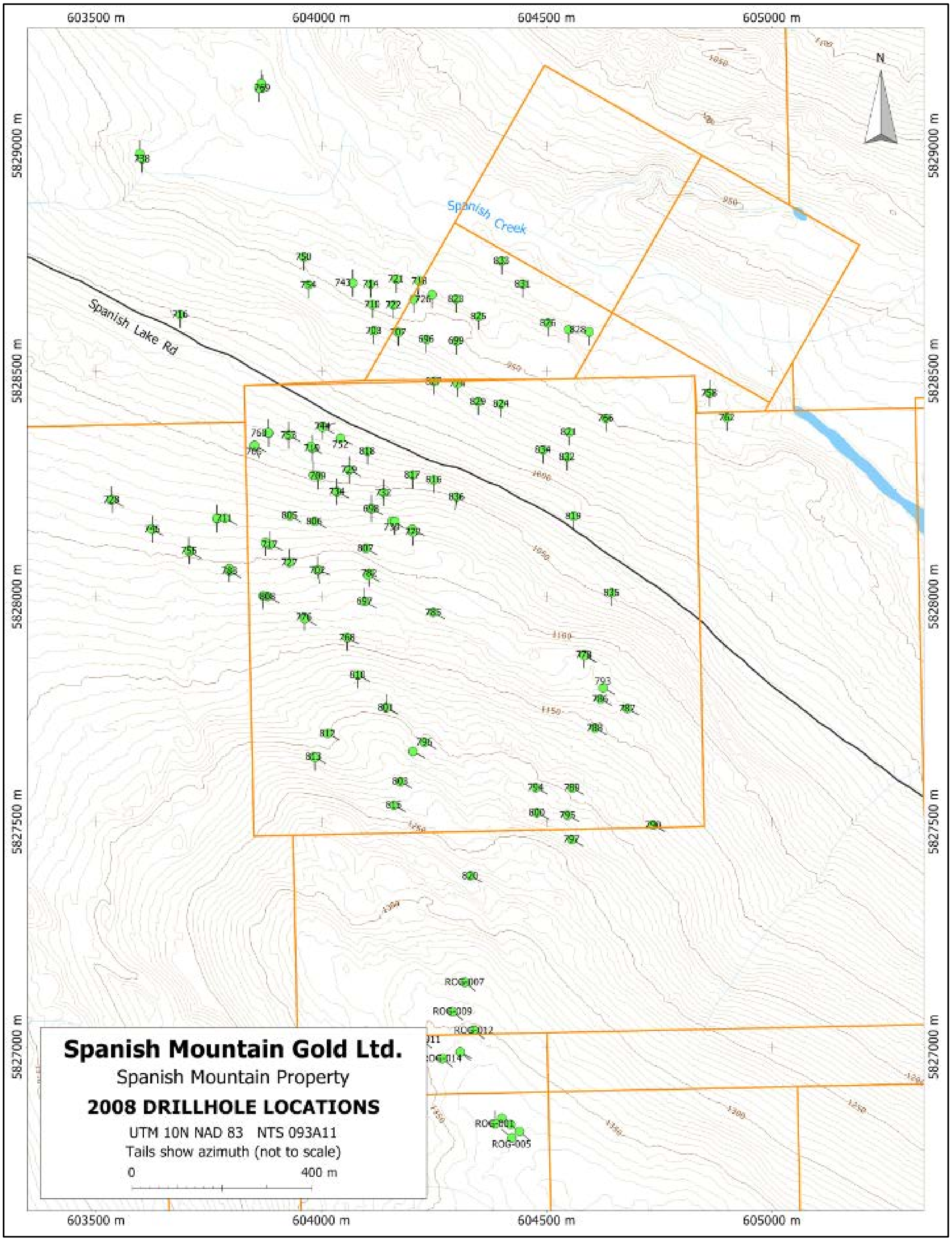


Figure 10-5 2008 Drillhole Locations

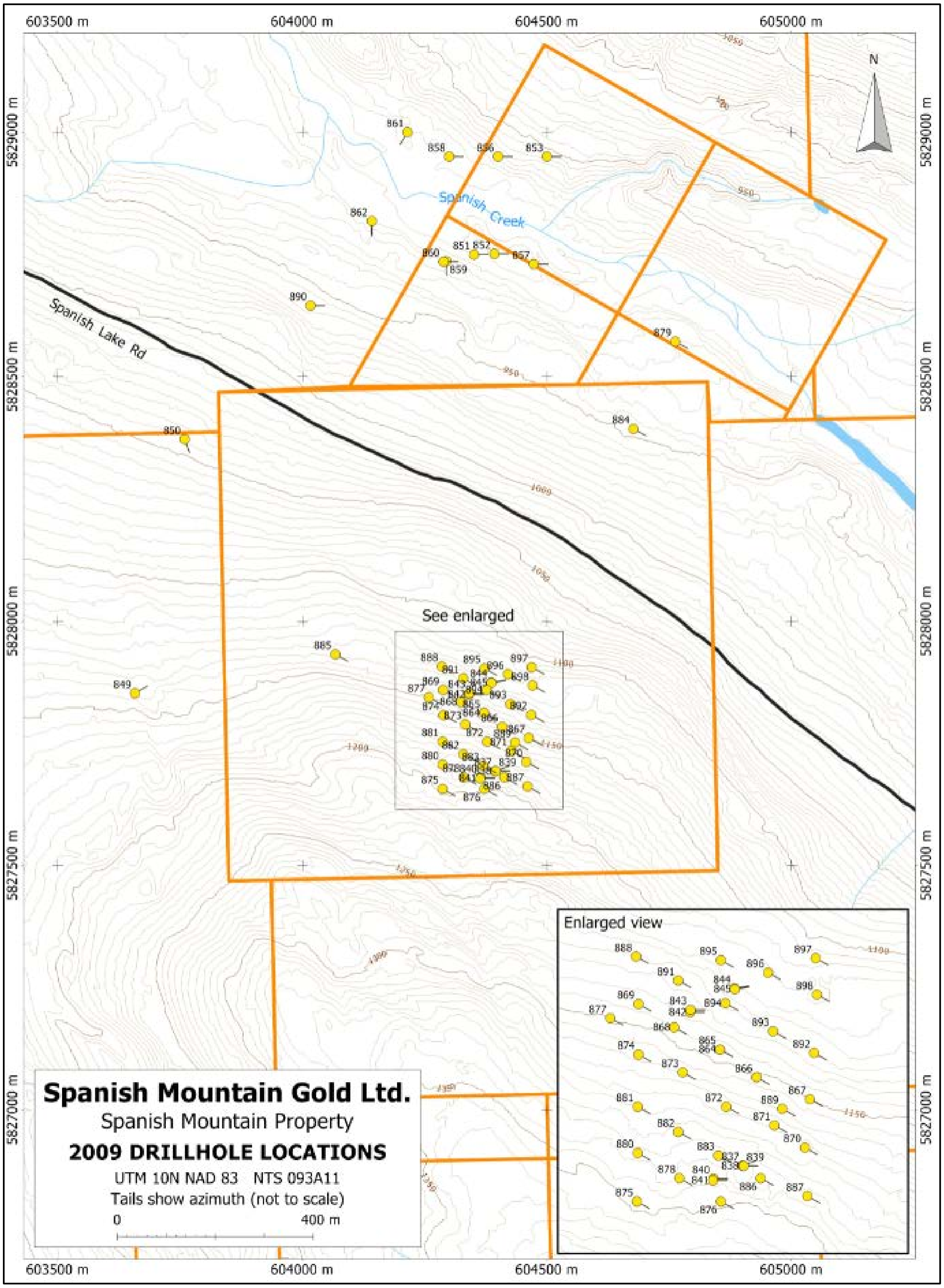


Figure 10-6 2009 Drillhole Locations

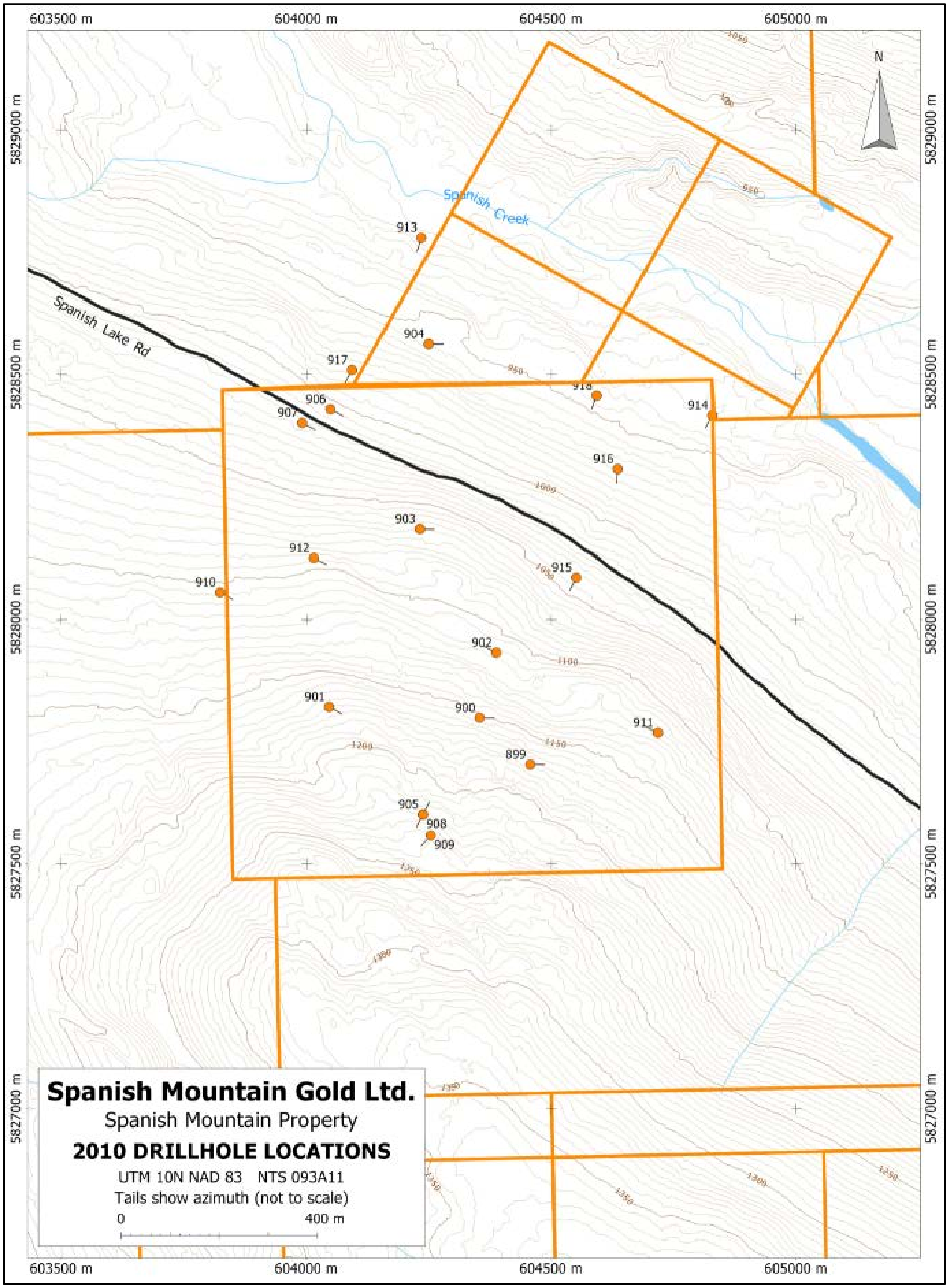


Figure 10-7 2010 Drillhole Locations

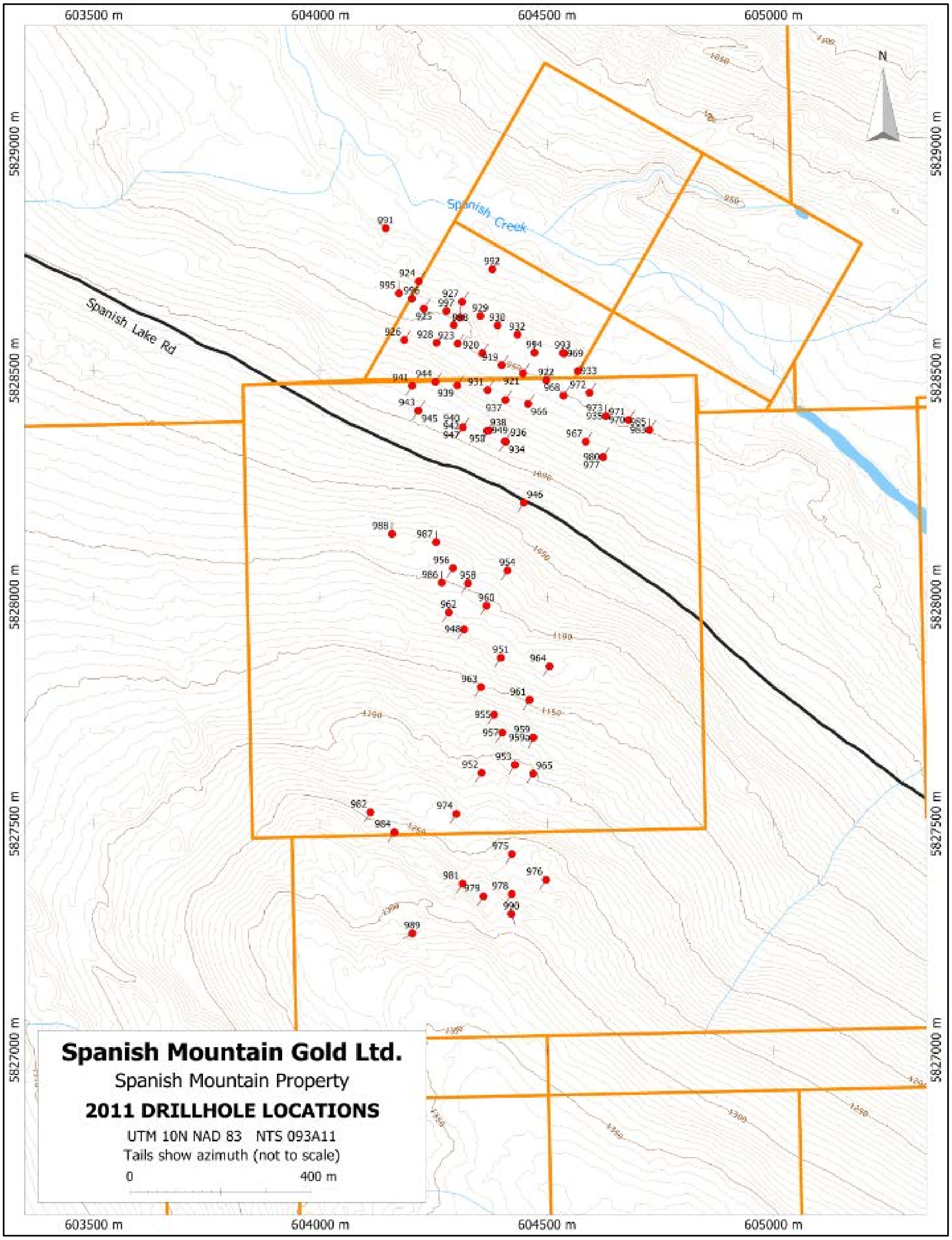


Figure 10-8 2011 Drillhole Locations

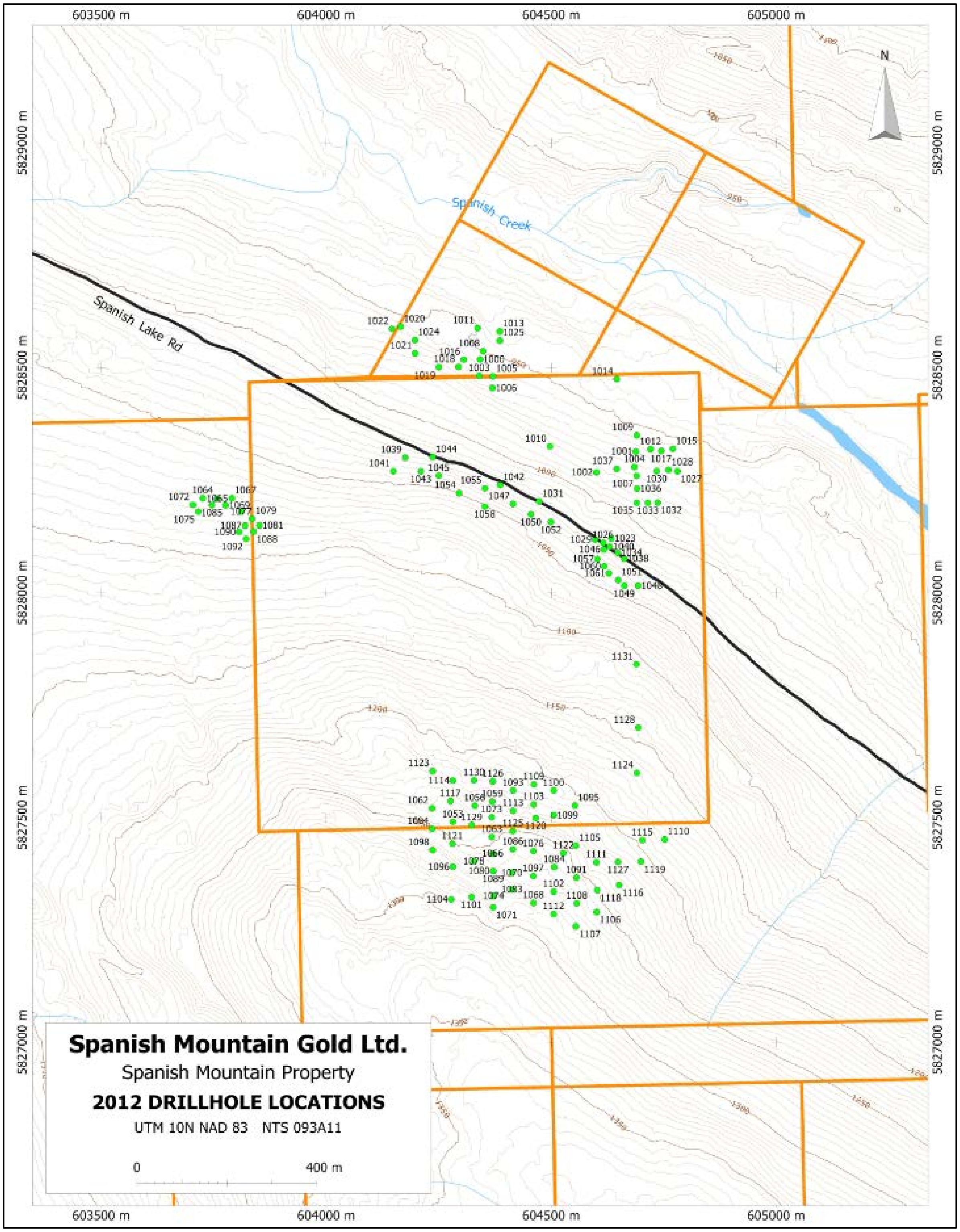


Figure 10-9 2012 Drillhole Locations

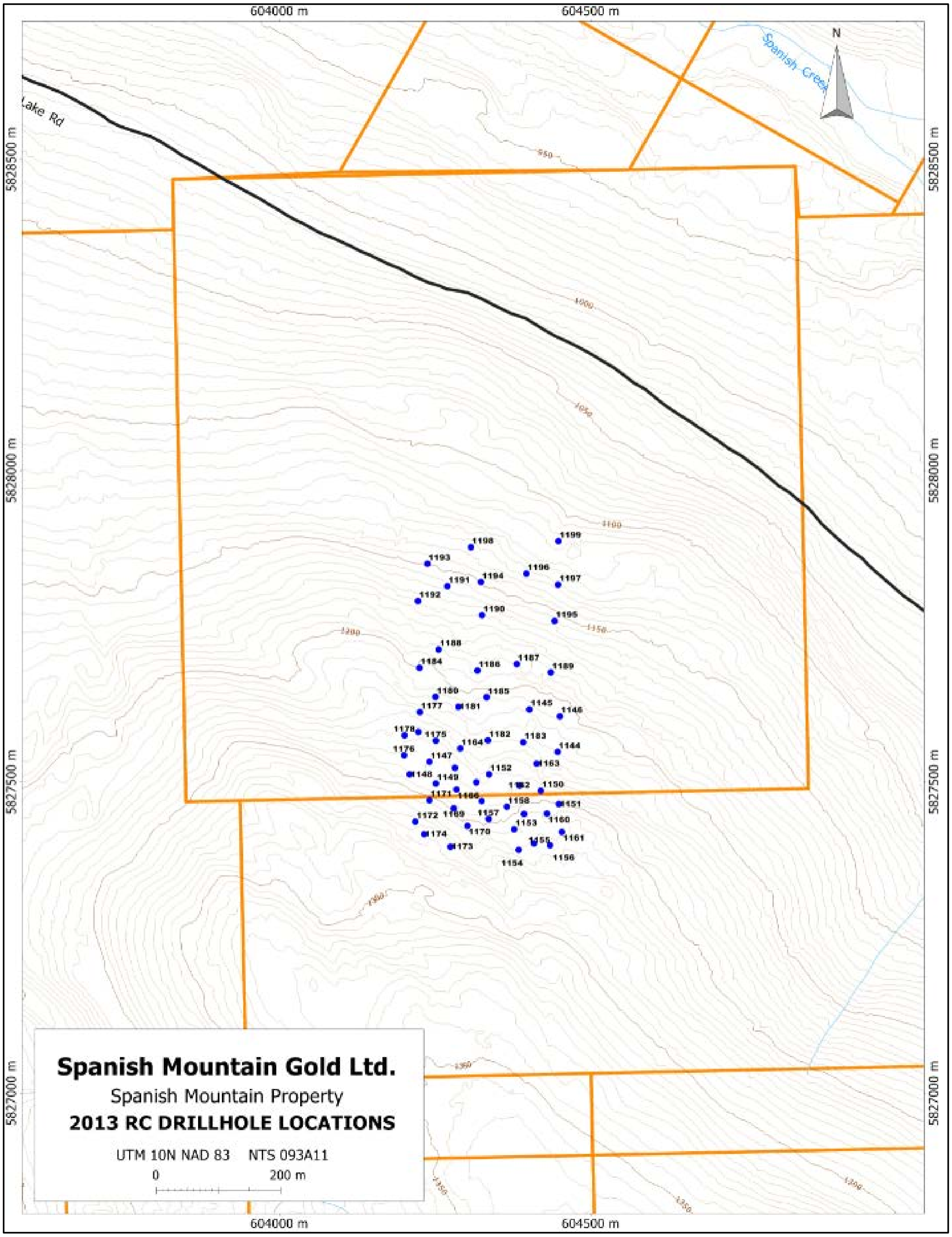


Figure 10-10 2013 Drillhole Locations

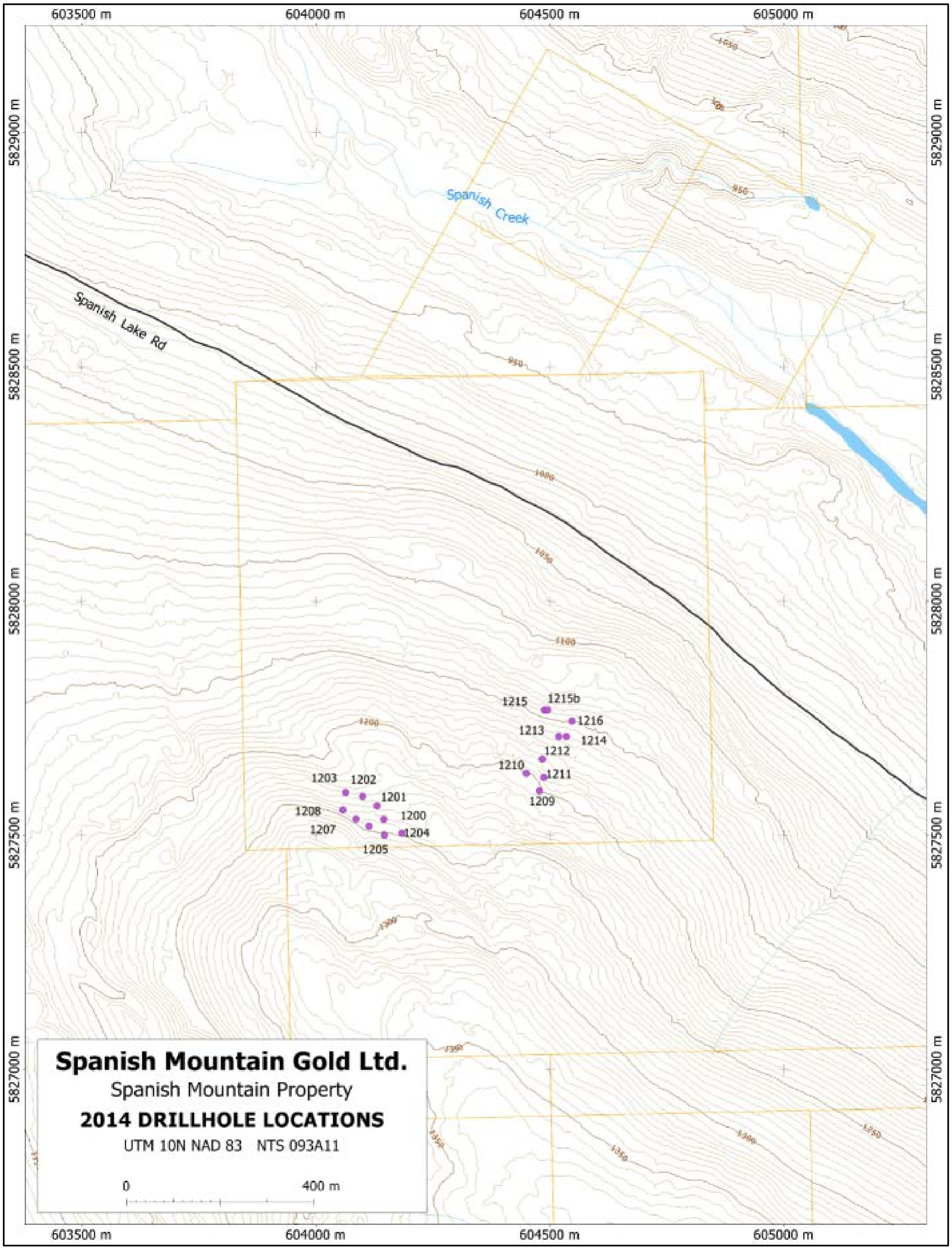


Figure 10-11 2014 Drillhole Locations

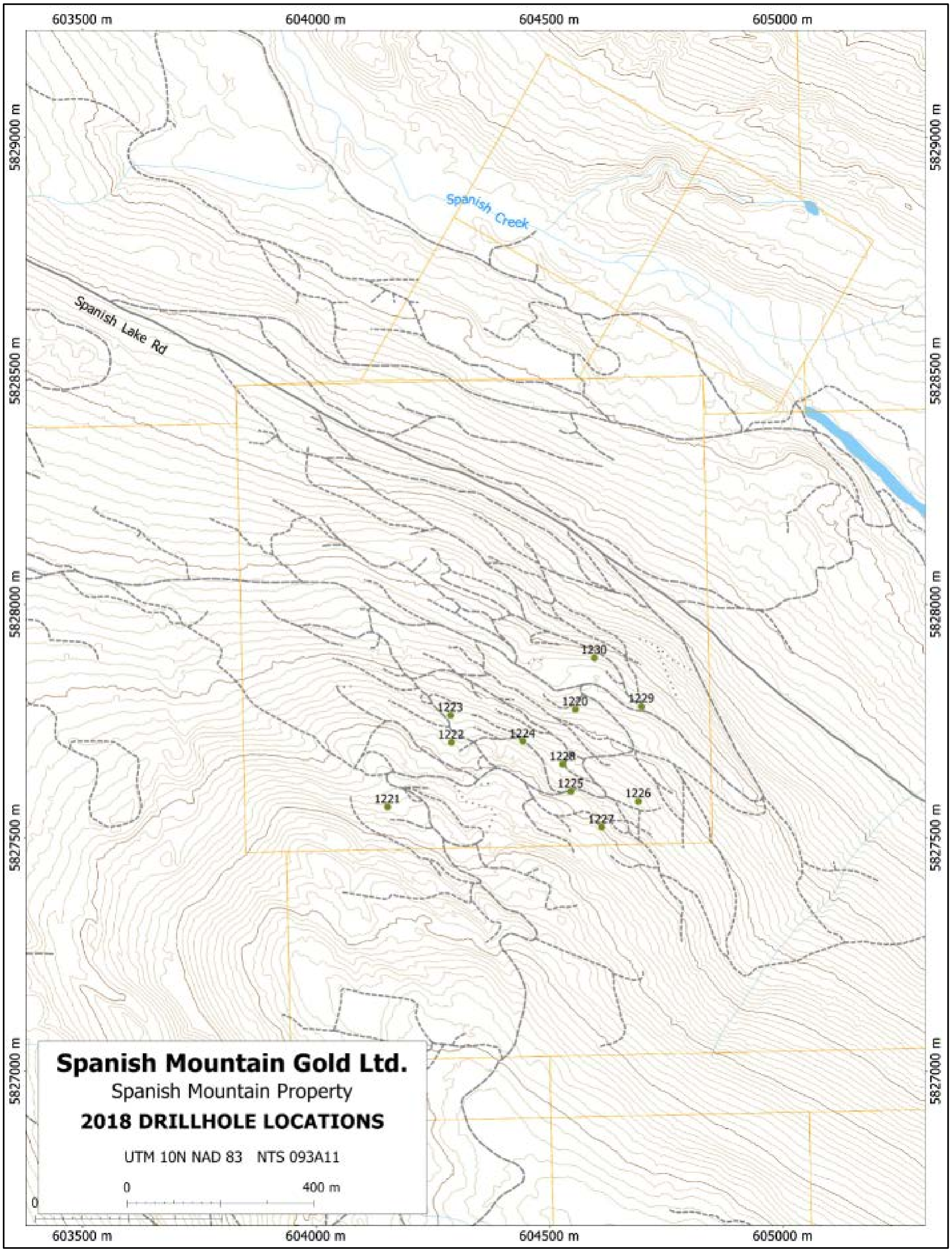


Figure 10-12 2018 Drillhole Locations

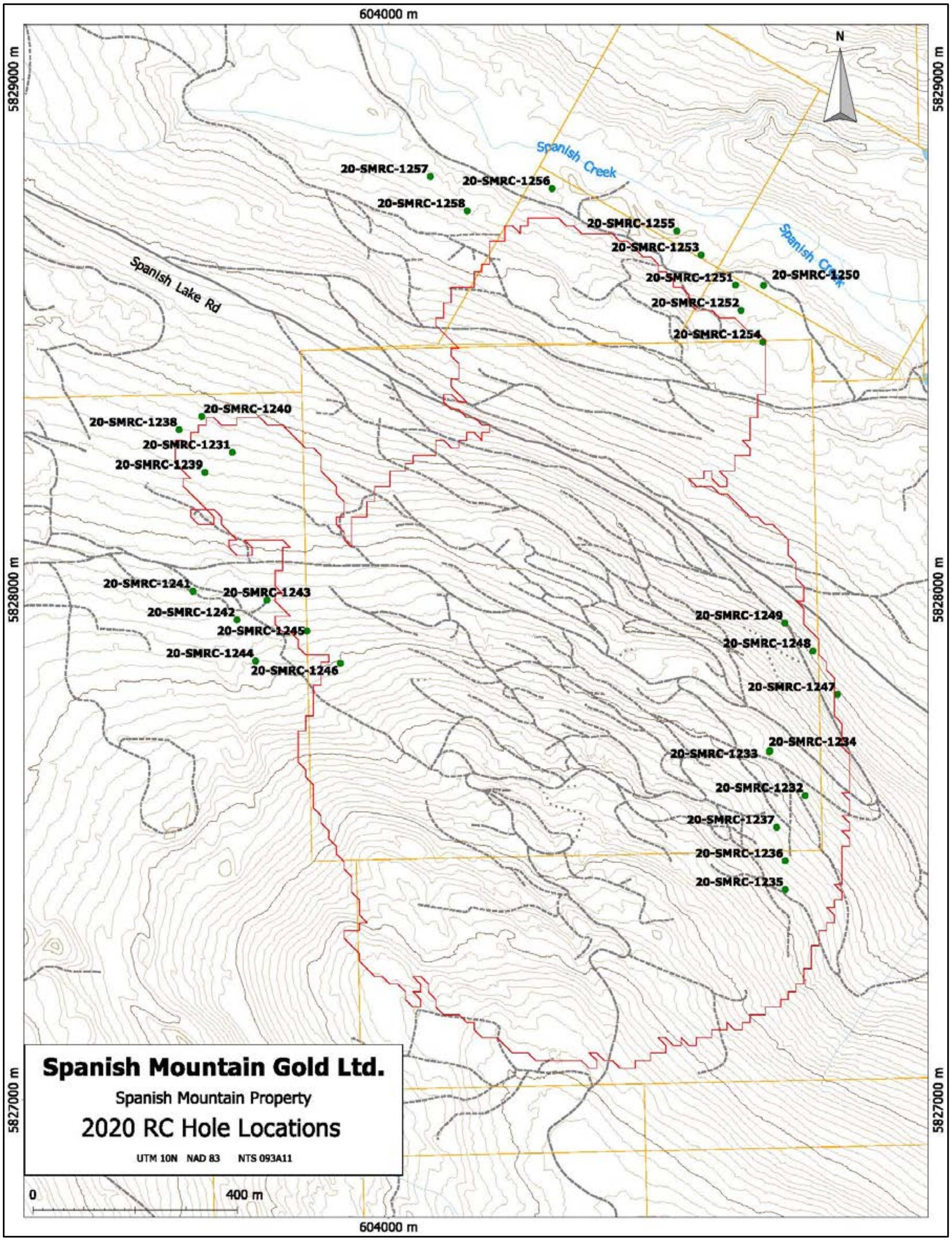


Figure 10-13 2020 RC Drillhole Locations

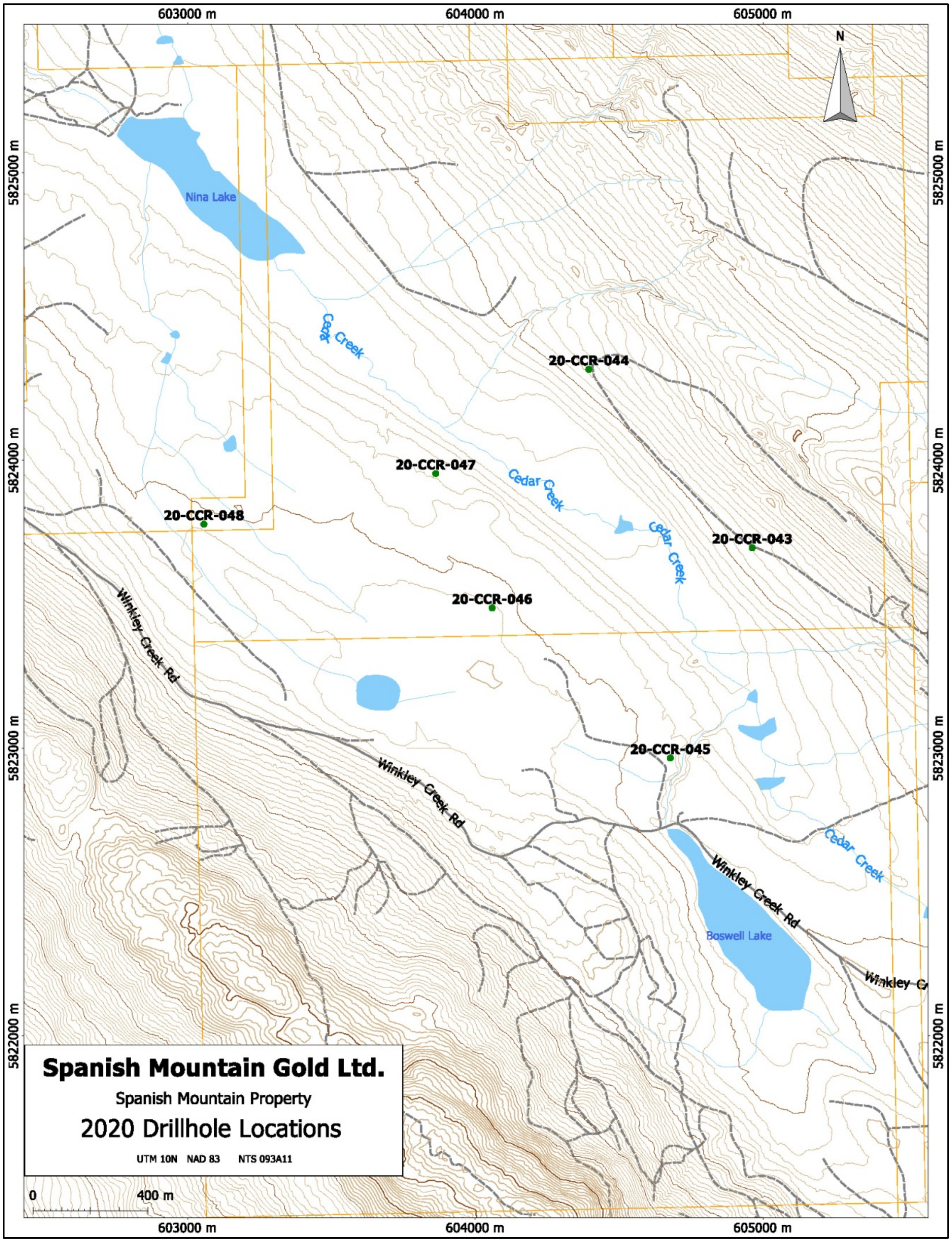


Figure 10-14 2020 Core Drillhole Locations

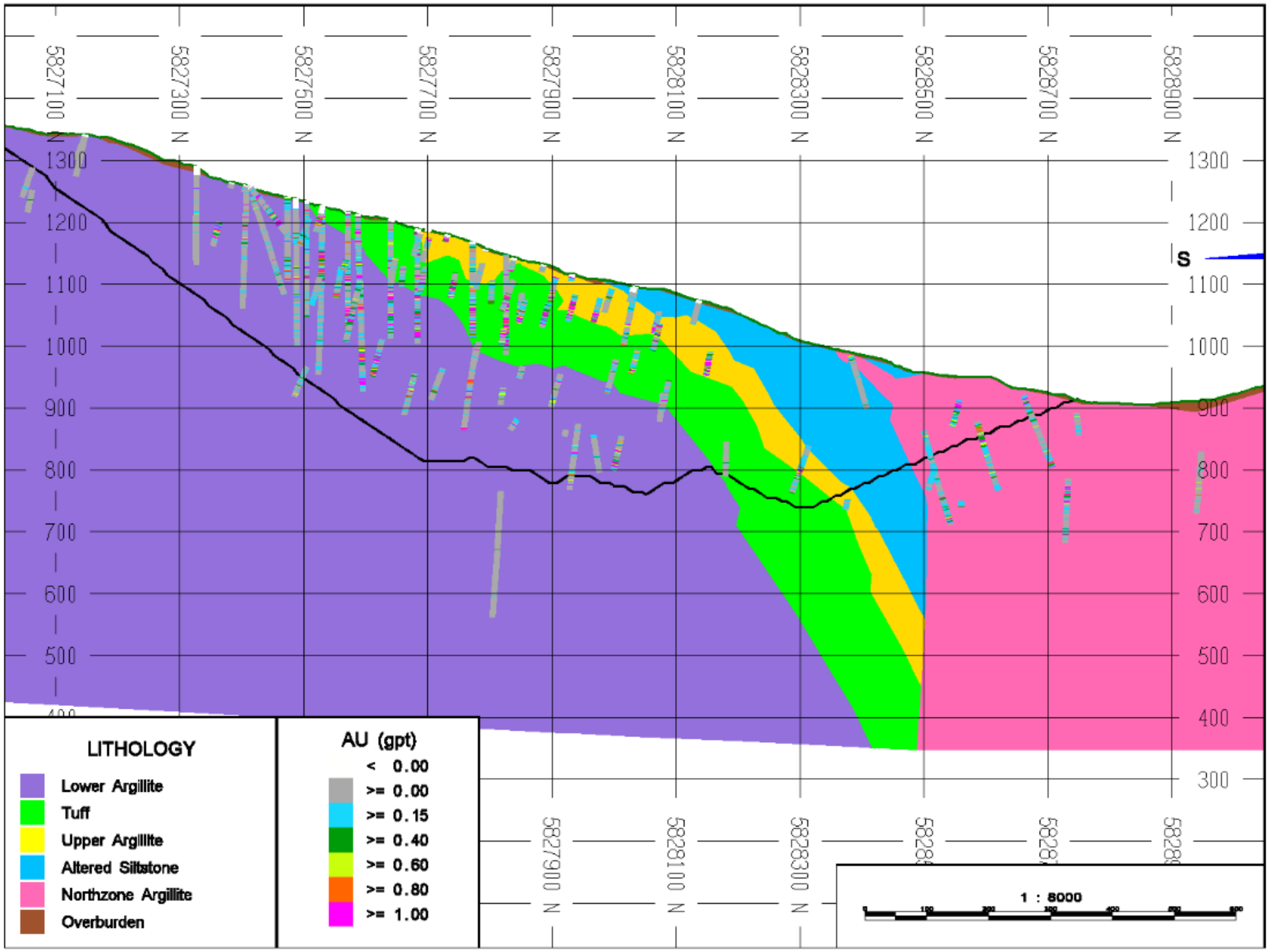


Figure 10-15 Section East 604325 looking West with Drill Grades (+/- 10 m from section) Relative to lithology and Resource Pit Shell

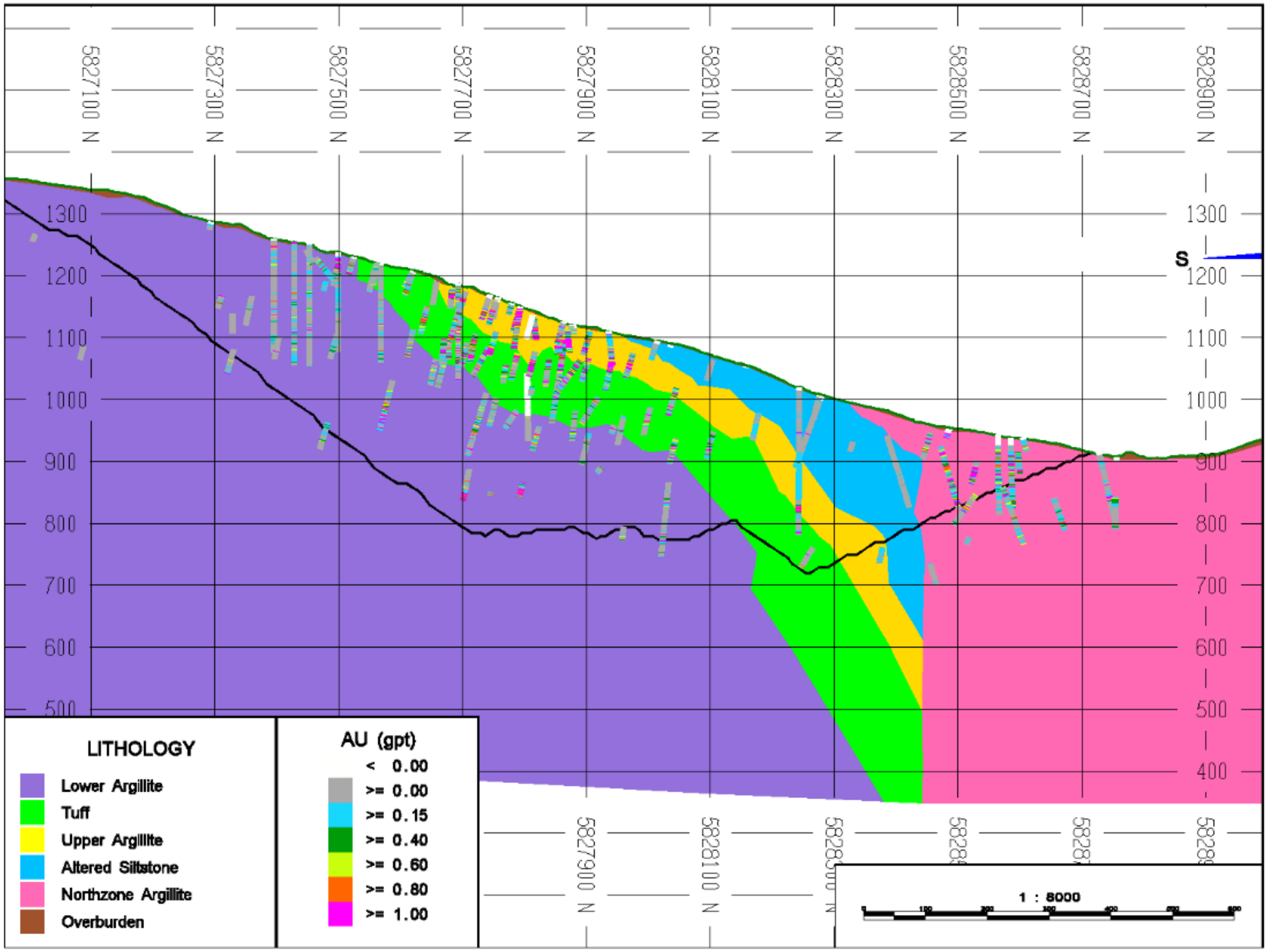


Figure 10-16 Section East 604385 looking West with Drill Grades (+/- 10 m from section) Relative to lithology and Resource Pit Shell

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

The following describes the sampling methods used by SMG in the 2010, 2011, 2012, 2018 and 2020 core drilling programs and in the 2013, 2014 and 2018 RC drilling program. Sampling methods are also described for the 2004 to 2009 programs. Other information in this Section was obtained from SMG, ALS Global Minerals Laboratory (ALS), and reports by co-author William Gilmour, P.Geo., who visited the Property on April 22, 2012, for the core drilling programs, on August 23, 2013, for the RC drilling program, and on September 12, 2019, and on December 2, 2020, for this report.

11.1 Sample Collection and Preparation

11.1.1 2004, 2005 and 2006 RC drilling

RC samples were collected every 1.52 m (5 feet) from the cyclone. Each sample was then run through a riffle splitter until the desired size was obtained. On the final split, both halves were bagged; one of these went to the laboratory for analysis and the other was retained as a similarly numbered reject, to be stored on site for further testing if required. The assay samples were closed with a plastic cable-lock and placed in similarly sealed rice sacks for shipment to the laboratory. The sacks were removed from the field nightly and stored at a secure staff facility. Samples for 2004 were shipped to Acme Laboratories in Vancouver via Van-Kam Freightways. For 2005 and 2006 drilling, the samples were shipped to Eco-Tech Laboratories in Kamloops.

The 2004 and 2005 RC drill programs were supervised by Robert Johnston, P.Geo.

11.1.2 2005 to 2009 Core Drilling

Core was taken from the drill site by pickup truck to the core handling facility. Core was logged for recovery and RQD and then geologically logged. Sample intervals, chosen by geologists, were normally 1.5 metres of core length, but shorter intervals were often used in areas of suspected higher grade, or where geological boundaries were encountered. The samples were cut in half lengthwise with a diamond saw and the sample portion was placed in plastic bags, along with assay tags, which were tied with plastic straps. All samples were stored in a secure staff facility until shipped to Eco-Tech Laboratories, via either staff, contact personnel, or by Van-Kam Freightways of Williams Lake, BC.

The 2005 core drill program was supervised by Robert Darney, P.Geo.

11.1.3 2010, 2011, 2012, 2018 and 2020 Core Drilling

The core was transported to SMG's core logging facility, where RQD procedures, core logging, and core sampling and splitting were carried out. The entire length of the core was sampled. Core was generally sampled in 1.5 m intervals, with shorter lengths used for lithology changes or the presence of visible gold. For the 2020 core drilling, most of the sample intervals are 2.0 m. Core splitting was done using diamond bladed rock saws operated by SMG personnel. Half of the core was sent for analysis; the other half was returned to the core box for a permanent record. Drill core samples were placed in plastic bags and shipped in rice bags through contract personnel (private courier) to ALS Minerals in Kamloops for sample preparation, followed by analysis in North Vancouver, BC.

The samples, together with QC/QA samples, were tabulated on batch sheets, with every sample batch comprising 80 samples. Each batch contained 4 blanks, 2 field duplicates, 4 standards, 2 samples scheduled to be made into laboratory duplicates and 68 core samples. The laboratory was instructed to process samples in single batches of 80 samples in numerical order, to assist with the QC/QA protocol.

11.1.4 2013, 2014, 2018 and 2020 RC Drilling

The RC drill program was designed with highest priority placed on careful and thorough sampling. A target depth of 200 m was used for each hole. Dry drilling was conducted above the water table. Once the water table was intersected, wet drilling techniques were required to complete the hole. Wet drilling entailed drilling while pumping both water and compressed air down the hole to operate the hammer and flush the drill cuttings back to surface.

Dry cuttings composed of rock chips and fine-grained powdered rock were blown to surface by compressed air and passed through a cyclone separator. Within the cyclone, the air was discharged out the top of the stack whereas the dry cuttings dropped into a 20-litre plastic pail placed directly beneath the cyclone.

The return cuttings were then transferred into an adjustable 50/50 riffle splitter having one-inch-wide shoots. One half of the material from the splitter was collected in a pre-labelled plastic sample bag; the other half was discarded. When a field duplicate was taken, the material from both sides of the riffle splitter was collected separately and sent for analysis.

To prevent cross-contamination between samples, compressed air was cycled through the rods to flush out all the cuttings at the end of a five-foot run. A by-pass valve allowed compressed air to also flush out any material left in the cyclone before drilling re-commenced for the next sample. The riffle splitter and pails were blown clean with forced air between samples. A skirt located directly above the drill bit helped seal the cuttings from escaping up the space between the rods and the sides of the drillhole, preventing loss of sample and contamination from possible wall rock caving.

Sample recovery was not quantified in the RC drilling; however, recoveries are likely very good. Some very fine particles were lost as airborne dust up the stack of the cyclone, but it is probable that the total weight of material lost as fine dust was << 0.5% of the weight of total returns.

Once a sample was collected, the bag was secured with a cable tie and loaded on a truck to be taken to the logging facility for further processing. Here the samples were weighed. Dry samples were shipped to the laboratory as received from the drill if they weighed <12 kg. Samples weighing over 12 kg were riffle split to achieve an appropriate target weight of 8 to 12 kg. The riffle splitting process is designed to produce the best possible, well mixed, representative sample for every five-foot interval drilled.

When the water table was reached in a drill hole and the hole started to produce significant amounts of water, the drillers switched over to wet drilling, which involved using both compressed air and water to drill and flush the cuttings to surface.

A Thompson wheel rotary splitter was used to split and collect the wet sample. To produce a sample similar in size to the dry samples, the adjustable splitter was set to produce 75% reject and 25% sample.

The water and the cuttings from the sample side of the splitter were collected in 20-litre plastic pails and transferred into larger 80-litre plastic tubs. When the tubs were 75% full, they were removed and a small amount of flocculent was added and mixed to help settle any suspended particulate matter in the water column. A few drops of dish soap were sometimes used to break the surface tension and sink particles floating on the surface; this was a more prevalent occurrence with samples containing graphitic argillite. Settling usually occurred within 2 to 3 minutes, at which time the water was decanted, and the fines transferred into a Micro-Por filter cloth sample bag designed to allow water to seep through, while retaining the fine material (-400 mesh). The cloth sample bags were hung on wooden racks near the drill to start the draining and drying process, then transported to the logging facility where they were hung to drip dry. The coarser cuttings settled in the 20-litre plastic pails were also transferred to a cloth bag and dried. Most wet-drilled samples consisted of 2 to 3 cloth bags.

Later in the season, when the weather became significantly colder, and decanting became difficult at the drill site, the water and cuttings were collected in the 20-litre plastic pails lined with plastic sample bags, secured with cable ties, and transported to the logging facility for processing indoors. Once dry, each sample, consisting of 2 to 3 labelled cloth bags, was placed in a labelled rice bag for shipment.

Chip trays were used to collect representative cuttings for each sample. A kitchen sieve was used to catch both dry and wet samples, which were collected from the reject side of the riffle splitter in the field. Larger chips were selected for ease of identification of rock type(s) present in the sample. The chips were placed in trays labelled with the sample and drill hole number and logged with the aid of a microscope.

Samples were shipped in batches containing 80 samples. Each batch of 80 samples contained 4 blanks, 2 field duplicates, 4 standards, 2 samples scheduled to be made into lab duplicates at the laboratory and 68 rock chip samples. Batches could contain either dry drilled samples, wet drilled samples (now dry) or a combination of both. The laboratory was instructed to process samples in single batches of 80 samples in numerical order to assist with QC/QA protocol. Samples with more than one bag of material were first dried as per the laboratory protocol before being mixed to produce a composite sample.

Sample preparation at the ALS laboratory involved drying the sample within the sample bag, then pouring it into trays, mixing, crushing, and sieving to 70% passing 10 mesh ASTM, pulverizing to 85% passing 75 µm or less.

11.2 Sample Preparation and Analysis

11.2.1 2004 and 2005 RC Drilling

For 2004 RC samples, analytical work was performed by Acme Laboratories, an ISO-certified laboratory, in Vancouver, BC. The RC chips were analyzed for metallic gold. The 500-g screen metallic method involved crushing the entire sample in an oscillating steel jaw crusher for 70% to pass -10 mm. A 500 g split was then pulverized and passed through a 150 mesh (100 µm grain size), producing a plus fraction (i.e., >100 µm) and minus fraction (i.e., <100 µm). In 2005, a 1000-g subsample was analysed. Two 30 g sub-samples of the finer screened material were analysed by fire assay, with an AAS finish. The entire amount of coarser material was also assayed by fire assay, with a gravimetric finish. The gold assays from the two fines were weight averaged, and this assay was then weight averaged with the assay from the coarser fraction, giving an overall assay for the sample. Multi-element analysis by ICP methods was also done.

11.2.2 2005 and 2006 Core Drilling

For core samples, analytical work was performed by Eco-Tech Laboratories, an ISO-certified laboratory, of Kamloops, BC. The entire half core sample was processed. Gold analysis and multi-element analysis were completed by the same methods as described in Section 11.2.1.

11.3 2006 RC Drilling

Eco-Tech carried out the analytical tests for the 2006 RC drilling. The RC chips were analyzed for metallic gold. Gold analysis and multi-element analysis were completed by the same methods as described in Section 11.2.1.

11.3.1 2007, 2008 and 2009 Core Drilling

For the 2007 to 2009 core drilling, Eco-Tech, Acme and ALS of Vancouver, BC, carried out the analytical tests. These laboratories were ISO certified. Gold analysis and multi-element analysis were completed by the same methods as described in Section 11.2.1.

11.3.2 2010, 2011, 2012, 2018 and 2020 Core Drilling

ALS carried out the analytical tests for the core drilling performed between 2010 and 2020. Gold analysis and multi-element analysis were completed by the same methods as described in Section 11.2.1. For the 2020 drilling, sample preparation comprised crushing the entire sample to 70% passing 2 mm, splitting off 250 g, and pulverizing the split to better than 85% passing 75 microns. The analysis was only for gold, using aqua regia extraction with ICP-MS finish on a 25 g subsample. The condemnation drilling demonstrated no mineralized zones, as 99 % of the samples assayed less than 0.05 g/t Au.

11.3.3 2013, 2014, 2018 and 2020 RC Drilling

ALS also carried out the analytical tests for the RC drilling performed between 2013 and 2010. Gold analysis and multi-element analysis were completed by the same methods as described in Section 11.2.1. As of the effective date of this report, all the 2020 RC samples had been received by the ALS preparation laboratory in Kamloops, but no analysis had commenced. Thus, no results from the 2020 RC drilling were available for use in this report.

11.3.4 Sample Security

Drill core/and RC cuttings were transported by SMG personnel to SMG's core logging facility, where rock quality designation (RQD) procedures, core logging, core splitting and core sampling were done. Also, at this facility, blank samples and standards were inserted into the sample stream. This facility is located on SMG's privately-owned property in the village of Likely, located about 7 km from the Main and North Zones. Core storage is also located there. Individual samples were placed in rice bags, which, in turn, were tied with plastic straps. Sample shipping was done through various companies, including Van-Kam Freightways, Canadian Freightways, or a private trucking courier to laboratories in Kamloops or Vancouver, BC. The security procedures meet quality control standards.

11.4 Quality Control and Quality Assurance Program

11.4.1 2004 and 2005 RC Drilling

For the 2004 and 2005 drilling programs, two different standards were inserted into the sample stream in the field. No information on the specifics of the standards is available. No other types of QC/QA samples were inserted.

Acme carried out in-house QC/QA analysis. Blank samples, standards and pulp duplicates were inserted and analysed, along with repeat analysis. No information on the blank analysis is available.

11.4.2 2005 Core Drilling

For the 2005 drilling, a comprehensive program of QC/QA was conducted. The field procedures included the insertion of analytical standards, sample blanks and preparation sample duplicates into the sample stream, at a rate of one for each 35 samples. The duplicate sampling procedure consisted of inserting a consecutively numbered, empty sample bag into the sample stream for later filling in the laboratory from a cut of the crushed sample. The specific procedures used for sample cutting are not known.

Eco-Tech carried out in-house QC/QA analysis on the 2005 samples. It is reported, by those supervising the program that the results of the SMG and the in-house programs were well within acceptable limits. However, information on the in-house QC/QA is not available to the authors of this report.

comprehensive program of QC/QA was conducted. The field procedures included the insertion of analytical standards, sample blanks and preparation sample duplicates, at a rate of one each 35 samples, into the sample stream. The duplicate sampling procedure consisted of inserting a consecutively numbered, empty sample bag into the sample stream for later filling in the lab from a cut of the crushed sampled. The specific procedures of the lab sample cutting are not known.

Eco-Tech carried out in-house QC/QA analysis on the 2005 samples. It is reported, by those supervising the program, that the results of the SMG and the in-house programs were well within acceptable limits. However, information on the in-house QC/QA is not available to the authors of this Report.

11.4.3 2006 RC Drilling

For the 2006 drilling, QC/QA comprised the insertion of analytical standards, sample blanks and preparation sample duplicates into the sample stream, at a rate of one each 35 samples. Two prepared standards, one higher and one lower grade, were obtained from CDN Laboratories of Delta, BC. There were also two other similar standards, but without any indication as to the average grade or the acceptable limits. Therefore, these results have not been evaluated.

As with the 2005 RC drilling, the duplicate sampling procedure consisted of inserting a consecutively numbered, empty sample bag into the sample stream for later filling in the laboratory from a cut of the crushed sample. The specific procedures used for sample cutting are not known.

11.4.4 2006 Core Drilling

No significant specific information on QC/QA for this drilling is available.

11.4.5 2007, 2008 and 2009 Core Drilling

For the 2007, 2008 and 2009 drilling, a comprehensive program of QC/QA was conducted. The field procedures included the insertion of analytical standards, sample blanks and preparation sample duplicates, at a rate of one each 35 samples. The duplicate sampling procedure consisted of inserting a consecutively numbered, empty sample bag into the sample stream for later filling in the laboratory from a cut of the crushed sample.

11.4.6 2010, 2011, 2012, 2018 and 2020 Core Drilling

Since December 2011, SMG has retained Discovery Consultants (Discovery) of Vernon, BC, to independently monitor the QC/QA procedures. This monitoring program did not constitute direction of any activities carried out by SMG staff. The monitoring was done under the supervision of William Gilmour, P.Geo., of Discovery. Discovery also provided Qualified Persons to monitor the core and RC drilling and sampling. QC/QA procedures carried out included the insertion into the sample stream by SMG of:

- field blank samples.
- empty bags with sample slips for insertion by the laboratory of duplicate preparation samples.
- field duplicate samples of core (the other half of the core).
- various gold standards (reference material).

In addition, ALS carried out its own in-house procedures for monitoring quality control, with the addition of its own laboratory blanks, pulp duplicates and standards.

Regarding QC/QA, field blanks were added randomly to the batches within every 30 samples.

Field standards consisted of five gold standards having varying gold content. One of three standards was added randomly within a group of 30 samples, with each standard added within every 90 samples.

At ALS, quality control samples used internally by the laboratory include control blanks, duplicates, and standards. The sample blank was inserted at the beginning of the batch, standards were inserted at random intervals, and duplicates were analyzed at the end of the batch.

11.4.7 2013, 2014 and 2018 RC Drilling

For the 2013, 2014 and 2018 drilling, the QC/QA procedures were the same as described in Section 11.4.6.

The quality control procedure to monitor possible contamination during the sample collection and preparation comprised the insertion of blank samples into the sample stream. Analysis of blank samples sent to ALS, within zones of gold mineralization in the sample stream, gave results within acceptable tolerances, demonstrating that no significant contamination during the sample preparation process.

The quality control procedure to measure the precision of the gold values involved the statistical treatment of duplicate pairs for RC cuttings, preparation (reject) and pulp samples. The 2013 RC drilling, as compared to the 2012 core drilling, shows a significant reduction in the variance in gold grade between duplicate samples. This is interpreted as being due to the significantly larger sample collected by the RC drilling, with both samples, with both RC and core samples being over the same 1.5 m sample interval.

The larger samples appear to have overcome some of the inherent difficulties when sampling heterogeneously distributed and coarse-grained gold.

The QC/QA protocol established for the currently advanced stage of exploration at Spanish Mountain was monitored by Discovery Consultants. At the core facility, a sub-batch was set at 20 samples, and four sub-batches were sent at a time to ALS for analysis. Each sub-batch included one field blank, one standard, and one duplicate, alternating between one field duplicate and one preparation duplicate.

Field standards consisted of gold standards having varying gold content. One of three standards was added randomly within a sub-batch of 20 samples, with each standard added within every 60 samples.

Field duplicates consisted of a second cut of crushed material taken at the laboratory. The sample bag with accompanying tag was added randomly within a group of 20 samples at the core facility and the material was added to the bag at the laboratory, prior to analysis. In effect, preparation duplicates are duplicates of the reject material. The preparation duplicate underwent both a second metallic screen determination for gold and a multi-element analysis.

At ALS, quality control samples used internally by the laboratory included analytical control blanks, pulp duplicates and standards. The analytical sample blank was inserted at the beginning of the batch, then every 40 samples. Two laboratory standards were inserted per 40 samples. Four laboratory standards were used for the metallic screen analysis and four other standards were used for the multi-elemental analysis. A pulp duplicate was assayed every 20 samples on the multi-element analysis.

The QC/QA results demonstrated no significant problems with the sample preparation or the sample analysis.

11.5 Contamination

Possible contamination during the sample preparation (crushing and pulverizing) was monitored by the insertion of blank samples into the sample stream in the field. The blank material was tested to ensure it was essentially devoid of gold.

11.5.1 2004 and 2005 Drilling

During the 2004 and 2005 RC drilling, as a precaution against contamination, the splitter and buckets were cleaned out between each sample, and the cyclone was also regularly checked and cleaned if required. No blank values are available for the 2004 drilling. For 2005, the inserted blank comprised dolomite.

For 2005, field inserted blank samples returned only one value (0.5% of the blank samples) greater than 0.02 g/t Au. The value is low, and the sample is not within a Mineral Resource area, so there is no issue of material contamination.

11.5.2 2006 Core and RC Drilling

As a precaution against contamination, the splitter and buckets were cleaned out between each sample, and the cyclone was also regularly checked and cleaned if required. The inserted blank comprised dolomite.

For 2006, field inserted blank samples returned only two values (0.5% of the blank samples) greater than 0.02 g/t Au. These samples are not within a Mineral Resource area, so there is no issue of material contamination.

11.5.3 2007, 2008 and 2009 Core Drilling

There were rare values (0.3% of the blank samples) greater than 0.03 g/t Au, but all were low and none within a Mineral Resource area, so there is no issue of material contamination.

11.5.4 2010, 2011, 2012, 2018 and 2020 Core Drilling

Field blanks consisted of sand collected from a gravel pit 30 km west of the Property. These samples, being sand, were not blind to the laboratory. In 2011, each 200-sample batch of blank sand was routinely checked by 15 samples sent for analysis at Eco-Tech. This sand was routinely found to be "clean" or devoid of gold mineralization. Note that this is naturally occurring sediment that is not guaranteed to always have very low gold values. Minor gold values are therefore likely to occasionally occur. For the 2011 program, the blanks were inserted randomly in the sample stream about every 30 samples.

For 2010, there were five values (2% of blank samples) greater than 0.02 g/t Au. The blank samples were low and were lower than their preceding sample. There was no issue of material contamination that would affect the resource.

For 2011 drilling, there were 2 values (0.3% of blank samples) greater than 0.02 g/t Au. There was no issue of material contamination.

During the 2012 program, blank samples were inserted into the sample stream at the rate of one every 20 samples; that is, 4 blank samples in each 80-sample batch. Repeat analysis of blank material sent to ALS within the sample stream gave results within acceptable tolerances, with only one sample (0.2% of blank samples) being less than the 0.05 g/t detection for metallic gold analysis, thus demonstrating no significant contamination during the sample preparation process.

11.5.5 2013, 2014 and 2018 RC Drilling

During the 2012, 2013 and 2014 programs, the samples were processed in-line within the laboratory, so that each sample followed the previous one consecutively. For the RC drilling, unlike processing of the core samples, where a blank can be inserted after a visible gold sample, the more immediate sampling procedures at the RC drill site did not allow for this.

The blank samples during the 2013 RC drilling returned more samples (5% of blank samples) containing anomalous gold, when compared to other years. However, these anomalous samples (up to 1.2 g/t Au) generally followed < 0.05 g/t Au samples; hence these blank values were inherent within the sample and were not as contamination from mineralized drill cuttings. They were also generally not near mineralized zones, hence there was no effect on resource estimation. Discussions with ALS have resulted in new procedures, which include the allocation of specific screens to this project and a more thorough cleaning of the screens between batches.

In the 2014 drilling, no blank samples exceeded the Au detection limit.

In the 2018 drilling, one blank sample happened to follow a high sample (25.7 g/t Au) and returned 1.1 g/t Au. However, it is the only such instance of possible contamination in all the SMG drilling. Therefore, it is concluded that there is no material effect on the resource estimate due to contamination during sample processing.

11.6 Precision

Duplicate sampling results are not an indication of analytical accuracy, but they indicate the natural variation in sampling. Accuracy is evaluated by the analysis of standards (Section 11.7). In this Section, statistical treatment was undertaken on suitable drill programs to gain an indication of precision by different drilling methods. It was demonstrated that the precision was better for RC drilling. Put another way, it was demonstrated that the variance was less for RC drilling.

11.6.1 2004 and 2005 Drilling

In 2005, a core hole was twinned with a 2004 RC hole, to compare the gold values in RC versus core drilling. In this instance it was found that when large intervals were weight averaged, the overall grade was similar. No statistical treatment was done to quantify the variance.

It is reported that several check analyses were carried out at Acme and ALS on sections of several drill holes. Minor variations occurred locally, but the overall Au grade did not change significantly. The underlying data are for these analyses are no longer available.

In 2004, no duplicate field or preparation samples were produced although it is reported that a few duplicate pulp samples were analysed by the laboratory.

Starting in 2005, preparation duplicates were produced by the laboratory, with the samples being placed in a pre-number sample bag inserted into the sample stream.

11.6.2 2007, 2008, 2009 and 2010 Core Drilling

A review of preparation duplicates was carried out. The correlation between original and duplicate samples was typical of the low precision found in core drilling programs on the Property.

11.6.3 2011, 2012, 2013, 2014 and 2018 Drilling

Duplicate field samples were prepared and analysed to measure precision. Precision is defined as the percent relative variation at the two-standard deviation (95%) confidence level. In other words, a result should be within two standard deviations of the mean, 19 times out of 20. The higher the precision number the less precise the results. Precision varies with concentration – commonly, but not always, the lower the concentration the higher the precision number. The precision values are determined from Thompson-Howarth plots (Smee, 1988). The duplicate sample results pair the original result with another sub-sample. This statistical method gives an estimate of the error in the process of sample collection, preparation, and analysis, thereby indicating the degree of homogeneity, or lack thereof, of gold within samples. Due to the relatively small number of duplicate samples in the 2014 and 2018 drilling, no precision figures were calculated.

Precision is a measure of the error in the analytical results from a variety of sources:

- core and RC cuttings sampling.
- sample preparation and sub-sampling.
- Analysis.

The three type of duplicates measure precision in the following processes:

- **core / RC cuttings duplicates:** the error in the sampling (splitting) of the core, in the sub-sampling of crushed and pulverized samples, and in analysis;
- **preparation (reject) duplicates**: the error in the sub-sampling of crushed and pulverized samples, and in analysis;
- **pulp duplicates**: the error in the sub-sampling of pulverized samples, and in analysis.

The core / RC cuttings duplicates and the reject (preparation) duplicates were inserted by SMG into the sample stream after the original sample.

Table 11-1 summarizes the estimated error in gold values for various duplicate samples.

Table 11-1 Summary of Sampling Errors (±%) for Various Duplicate Samples

Au g/t	0.20	0.50	1.00
Core, 2012	21	42	49
RC cuttings, 2013	19	16	15
Reject Core, 2011	21	17	16
Reject Core, 2012	16	14	13
Reject RC cuttings, 2013	15	15	16
Pulp core, 2010 to 2012	24	12	8
Pulp RC cuttings, 2013	15	6	3

11.6.4 2012 to 2018 Core/RC Cuttings Duplicates

There were no core duplicates for pre-2012 drilling. For the 2012 core drilling program, duplicate core samples (the other half of the split core) were inserted into the sample stream at the rate of one every 40 samples (427 pairs); that is, 2 duplicate samples in each 80-sample batch.

Sample pairs containing an average grade of at least 0.06 g/t Au (202 pairs) were plotted by the Thompson-Howarth method. These duplicate samples underwent the same metallic gold analysis as did the regular samples. The results are summarized in Table 11-2.

Table 11-2 2012 Core Duplicates – Precision Values

Precision Values (%), n=202				
Au g/t	0.20	0.50	0.75	1.00
	42.2%	83.6%	92.8%	97.4%

At the 95% confidence level, the precision values indicate about a ±21% error for 0.20 g/t Au values and about a ±42% error for 0.50 g/t Au values. This is the total error for core sampling, sub-sampling of crushed and pulverized core, and analysis.

In the 2013 RC program, samples were inserted into the sample stream at the rate of one every 40 samples (175 pairs); that is, 2 duplicate samples in each 80-sample batch.

For the dry drilling, when a field duplicate was taken, the material from both sides of the riffle splitter was collected and sent for analysis. For the wet drilling, the wheel splitter was changed to a 50/50 split with both sides being collected. Sample pairs containing an average grade of at least 0.06 g/t Au (110 pairs) were plotted by the Thompson-Howarth method. These duplicate samples underwent the same metallic gold analysis as did the regular samples. The results are summarized in Table 11-3.

Table 11-3 2013 RC Cuttings Duplicates – Precision Values

Precision Values (%), n=110				
Au g/t	0.20	0.50	0.75	1.00
	38.0%	31.3%	29.8%	29.0%

At the 95% confidence level the precision values indicate about a ±19% error for 0.20 g/t Au values and about a ±16% error for 0.50 g/t Au values. This is the total error for cuttings sampling, sub-sampling of crushed and pulverized cuttings, and analysis.

11.6.5 Reject Duplicates

For the 2011 drilling used in the 2011 Resource Estimate, the laboratory systematically produced, every 30 samples (901 pairs), a duplicate sample from the saved reject (crushed) core. Sample pairs containing an average grade of at least 0.04 g/t Au (418 pairs) were plotted by the Thompson-Howarth method. These duplicate samples underwent the standard fire assay gold analysis on the -150 mesh (<100µm) pulp. The results are summarized in Table 11-4.

Table 11-4 2011 Core Reject Duplicates – Precision Values

Precision Values (%), n=418				
Au g/t	0.20	0.50	0.75	1.00
	41.6%	34.3%	32.6%	31.8%

At the 95% confidence level the precision values indicate about a ±21% error for 0.20 g/t Au values and about a ±17% error for 0.50 g/t Au values. This is the total error for sub-sampling of crushed and pulverize core, and for analysis.

For the late 2011 and the complete 2012 drilling, SMG selected samples, one in every 40 (492 pairs), for a duplicate sample; that is, 2 samples in each 80-sample batch. An empty bag with a sample slip was inserted into the sample stream and ALS filled the bag with a duplicate sample from the crushed core. These duplicate samples underwent the same screen metallic gold analysis as did the regular samples.

Sample pairs containing an average grade of at least 0.06 g/t Au (209 pairs) were plotted by the Thompson-Howarth method. The results are summarized in the following Table 11-5.

Table 11-5 2012 Core Reject Duplicates – Precision Values

Precision Values (%), n=209				
Au g/t	**0.20**	**0.50**	**0.75**	**1.00**
	31.6%	27.0%	26.0%	25.4%

At the 95% confidence level, the precision values indicate about a ±16% error for 0.20 g/t Au values and about a ±14% error for 0.50 g/t Au. This is the total error for sub-sampling of crushed core (preparation) and pulverized core, and analysis.

For the 2013 RC drilling, SMG also prepared one duplicate in every 40 samples (173 pairs), using the same procedure as described above. These duplicate samples underwent the same screen metallic gold analysis as did the regular samples.

Sample pairs containing an average grade of at least 0.06 g/t Au (106 pairs) were plotted by the Thompson-Howarth method. The results are summarized in Table 11-6.

Table 11-6 2-13 RC Reject Duplicates – Precision Values

Precision Values (%), n=106				
Au g/t	**0.20**	**0.50**	**0.75**	**1.00**
	29.2%	30.6%	30.9%	31.1%

At the 95% confidence level the precision values indicate about a ±15% error for 0.20 g/t Au values and about a ±15% error for 0.50 g/t Au. This is the total error for sub-sampling of crushed core (reject or prep) and pulverized core, and analysis.

11.6.6 Pulp Duplicates

At the 95% confidence level, the precision values indicate about a ±15% error for 0.20 g/t Au values and about a ±15% error for 0.50 g/t Au (Table 11-7). This is the total error for sub-sampling of crushed core (reject or prep) and pulverized core, and analysis.

Table 11-7 2010 – 2012 Core Pulp Duplicates – Precision Values

Precision Values (%), n=7,278				
Au g/t	**0.20**	**0.50**	**0.75**	**1.00**
	48.6%	23.4%	18.3%	15.6%

At At the 95% confidence level, the precision values indicate about a ±24% error for 0.20 g/t Au values, a ±12% error for 0.50 g/t Au values and a ±8% error for 1.00 g/t Au values. This is the error for the sub-sampling of the pulverized core (pulp), and analysis. Note that the pulp samples exclude the coarser metallic gold.

For the 2013 RC drilling, ALS prepared two 30 g sub-samples for every sample of core, producing 5,937 pairs. Sample pairs containing an average grade of at least 0.04 g/t Au (4,092 pairs) were plotted by the Thompson-Howarth method. The results are summarized in Table 11-8.

Table 11-8 2013 RC Pulp Duplicates – Precision Values

Precision Values (%), n=4,092				
Au g/t	0.20	0.50	0.75	1.00
	29.8%	11.9%	8.0%	6.0%

At the 95% confidence level the precision values indicate about a ±15% error for 0.20 g/t Au values, a ±6% error for 0.50 g/t Au values and a ±3% error for 1.00 g/t Au values. This is the error for the sub-sampling of the pulverized core (pulp), and analysis. Note that the pulp samples exclude the coarser metallic gold.

11.7 Accuracy

The analytical accuracy was evaluated by inserting standards (also called reference material) into the sample stream. The standards have an expected value with a minimum and maximum range. The range is based on two standard deviations from the average. This means that 19 times out of 20 the values should be within the range. Conversely it also means that 1 time out of 20 the value could exceed the expected range.

11.7.1 2004 Drilling

In 2004, two different standards were inserted into the sample stream. No information on the specifics of the standards is available. No other types of QC/QA samples were inserted.

Acme carried out its own in-house QC/QA analysis. Blank samples, standards and pulp duplicates were inserted and analysed, along with repeat analysis. It was reported that no QC/QA problems were noted.

11.7.2 2005 Drilling

In 2005, two different standards were inserted into the sample stream. For core and RC samples, only one sample exceeded the acceptable limits for each standard. No abnormal trends or material bias were noted.

11.7.3 2006 Drilling

In 2006, two different standards were inserted into the sample stream. Only one sample was outside of the acceptable limits. No abnormal trends or material bias were noted.

11.7.4 2007 Drilling

In 2007, three different standards were inserted into the sample stream. Only one sample was outside of the acceptable limits. No abnormal trends were noted. The highest-grade standard had a low bias, but except for one sample, was within acceptable limits. Most of the drill samples are significantly lower than the higher standard value, so the low bias is not of material significance.

11.7.5 2008 Drilling

In 2008, three different standards were inserted into the sample stream. Only three samples were outside of the acceptable limits. No abnormal trends or material bias were noted.

11.7.6 2009 Drilling

In 2009, six different standards were inserted into the sample stream. Samples were analysed by Eco-Tech or ALS. The results from ALS show more variance than those from Eco-Tech. However, no abnormal trends or material bias were noted.

11.7.7 2010 to 2020 Drilling

All but one of the SMG inserted gold standards used between 2010 and 2020 were produced by CDN Resources Labs Ltd (CDN) of Langley, BC, and were certified to 2 standard deviations by a certified assayer and by a professional geochemist. One standard was produced by Ore Research & Exploration of Australia.

In the 2010 and 2011 core drill programs, one of several standards was inserted randomly about every 30 samples. For the 2010 drilling, standards were submitted with expected grades of 0.39, 0.78, 1.16 and 4.83 g/t Au and, for the 2011 drilling, standards had expected grades of 0.21, 0.39, 0.78, 1.14, 1.16 and 3.77 g/t Au.

In the 2012 core drilling, standards were inserted into the sample stream at the rate of one every 20 samples; that is, 4 standard samples in each 80-sample batch. During this program, some CDN standards were replaced, as others were depleted, with ones of similar grade. In total, 7 different standards were used with expected grades of 0.34, 0.41, 1.14, 1.47, 1.97, 2.71 and 3.77 g/t Au.

In the 2013 RC drilling, standards were again inserted into the sample stream at the rate of one every 20 samples. In total, 5 different standards were inserted by SMG with expected grades of 0.34, 1.44, 1.97, 3.18 and 3.77 g/t Au. The results of 4 standards inserted by ALS were also monitored.

In the 2014 RC drilling, standards were also inserted into the sample stream at the rate of one every 20 samples. In total, 4 different standards were inserted by SMG with expected grades of 0.34, 1.44 and 3.18 g/t Au. The results of 4 standards inserted by ALS were also monitored.

The QA monitoring included plotting the results for each SMG and ALS standard. The charts were regularly reviewed for results outside of the expected values ranges. No abnormal trends or material bias were noted. Minor re-analysis of a group of samples was done, no changes in the original results were warranted.

11.8 Comments on Section 11

It is the opinion of the QP, William Gilmour, P.Geo., that the sample security, sample preparation and analytical procedures during the exploration programs followed accepted industry practice appropriate for the stage of mineral exploration undertaken and are NI 43-101 compliant.

12 DATA VERIFICATION

The 2004 RC drilling program was carried out by SMG's joint venture partner at the time, Wildrose Resources Ltd, under the supervision of Robert Johnston, P.Geo., of Mincord Exploration Consultants. The 2005 core and RC drilling program by SMG was conducted under the supervision of Robert Darney, P.Geo., of Pamicon Developments Ltd.

The 2006 to 2009 drilling programs by SMG were completed under the direction of Robert Singh, P.Geo., of Pamicon.

The 2010 core drill program was carried out by SMG under the supervision of Judy Stoeterau, P.Geo., of SMG.

The 2011 and 2012 core drill programs pertaining to the Mineral Resource Estimate were carried out by SMG under the supervision of Judy Stoeterau, P.Geo., of SMG.

The 2013, 2014 and 2018 RC drill and core drill programs were carried out by SMG under the supervision of Judy Stoeterau, P.Geo., of SMG. Qualified Persons from Discovery Consultants monitored the drilling, sampling and QC/QA procedures, and reviewed analytical certificates throughout the drill program. The co-author of this report, William Gilmour, P.Geo., was responsible for reviewing the results, including QC/QA, but at no time directed the activities of SMG staff.

For the 2011 and later drill programs, co-author Gilmour obtained the analytical results directly from ALS. The data included CSV files and PDF files. There were no discrepancies noted between the two types of data files. By using the sample templates produced in the field, the sample numbers for specific intervals of drill core or cuttings were matched up with the sample numbers in the analytical reports. The same procedure was used for field blank samples, field duplicate samples and field standards, producing a compiled spreadsheet of the all the results.

An independent validation of the drill hole database was carried out by Ginto Consulting Inc. prior to the estimation of the Mineral Resources for this report. In this exercise, approximately 10% of the gold assays from the drill hole database were checked against the original assay certificates. Similarly, approximately 10% of the drill hole collar coordinates and downhole surveys from the drill hole database were checked against the drill logs. Overall, no significant errors were found, and the drill hole database was deemed valid for the estimation of Mineral Resources.

During the drill hole database validation exercise, a survey of drill hole collars carried out in 2008 by Allnorth Consultants Limited indicated different collar coordinates for 48 holes. From further investigation, it was found that these holes were re-surveyed after this initial survey and that the collar coordinates in the drill hole database are from this re-surveying program, reflecting the correct collar coordinates for these holes. Field checks for a few drill hole coordinate surveys were carried in December 2020 and have confirmed the current coordinates. It is recommended that all the 48-hole collars be re-surveyed during the 2021 summer months and documented to provide a reference for these holes.

12.1 Comments on Section 12

All the data used in the Mineral Resource Estimate for this report has been reviewed by QP Marc Jutras, P.Eng., M.A.Sc., and in his opinion sufficient verification checks have been undertaken on the drill hole database to provide confidence that the database is of sufficient quality to support the Mineral Resource Estimate.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

The Spanish Mountain Gold deposit has been the subject of several metallurgical testwork programs that were focussed on comminution, gravity, flotation, and cyanidation, as described in Section 13.2.

During the 2019 PEA, the process flowsheet comprised of:

- Primary, secondary, and tertiary crushers
- Ball mill grinding
- Primary grind (P80 180 µm);
- Rougher flotation, 1st Cleaner, and 2nd Cleaner flotation
- Cleaner tailings gravity concentration
- Cyanide leaching, adsorption in a carbon-in-leach (CIL) circuit
- Cyanide destruction.

During the 2020 PFS, the flowsheet design was modified to a simpler and lower cost comminution and flotation circuit and to optimize the recovery comprising:

- Primary crusher
- SABC grinding & classification circuit,
- Primary grind (P80 180 µm)
- Primary Gravity Concentration and ILR
- Rougher flotation, 1st Cleaner, and 2nd Cleaner flotation (DFR Flotation cells)
- Cleaner tailings gravity concentration
- Cyanide leaching, adsorption in a carbon-in-leach (CIL) circuit
- Cyanide destruction.

13.2 Metallurgical Testwork

For the 2020 PFS, metallurgical test work conducted by SGS, G&T, Met-Solve, Knelson, and McClelland Laboratories was reviewed. Relevant historical testwork programs and testwork used as the basis for the current flowsheet are presented in Table 13-1 (Summary of Historical Testwork) and Table 13-2. (Summary of Metallurgical Testwork Scope).

Table 13-1 Summary of Historical Testwork

Type of Test Work	Laboratory	Date
Report on Metallurgical Testing – SM Drill Core Composites MLI Job. No.4373	McClelland Laboratories, Inc	November 20, 2019
MS1735 SMG project	(Project MS1735) Met-Solve Laboratories, Inc	March 26, 2017
Metallurgical testing on variability samples from the SMG project	(Project KM3185) by G&T Metallurgical Services	June 21, 2012
Preliminary sighter setting and filtration testing from the SMG project	(Project KM3403) by G&T Metallurgical Services	May 3, 2012
An investigation into a variability test program on samples from the SMG deposit	(Project 12488-002 – Report 2) by SGS Canada	March 19, 2012
A metallurgical test program on samples from the SMG deposit	(Project 12488-002 – Report 1) by SGS Canada	March 7, 2012
The HPGR Grindability Characteristics of Two Samples	(Project 12488-003 – Final Report by SGS Canada	December 20, 2011
Gravity Modelling Report SMG	(Project 20559-1) by Knelson Research & Technology Centre	Oct.18, 2011
An investigation into grinding circuit design for the SMG project based on small-scale data	(Project. No. 12488-001) by SGS Canada	Dec. 23, 2010
Metallurgical testing on samples from the SMG project	(Project KM2637) by G&T Metallurgical Services – Preliminary and Final reports	Aug.30, 2010 Sept 2, 2011
Metallurgical Test report SMG	(Project KRTC 20559) by Knelson Research & Technology Centre	May.19, 2010
Comparative gold content in the core using gravity concentration techniques – SMG project	(Project KM2538) by G&T Metallurgical Services	Apr.9, 2010

Table 13-2 Summary of Scope of Relevant Metallurgical Testwork

Project	Sample ID	Type of Rock	Head Assays	Comminution RWI & BWI	JK DWT	E-GRG	Gravity Separa-tion	Rougher Flotation	Cleaner 2	Scv. Grav. Conc. on Clnr. Tails	Cyanidation on Clnr Conc. Std Bottle	Grav. Conc. Intensive Cyanidation	Cyanide Destruc-tion
G&T (Project KM2637)	865-1	Siltstone*	X	X	--	--	X	X	--	--	X	--	--
	865-2	Argillite	X	X	--	X	X	X	--	--	X	--	--
	865-3	Siltstone*	X	X	--	X	X	X	--	--	X	--	--
	871	Argillite/Siltstone	X	--	--	--	X	X	X	--	X	--	--
	871B	Siltstone*	X	--	--	--	X	X	--	--	X	--	--
	872A	Argillite	X	--	--	--	X	X	X	--	X	--	--
	872B	Argillite/Siltstone	X	--	--	--	X	X	--	--	X	--	--
	891	Siltstone/Argillite	X	--	--	--	X	X	X	--	X	--	--
	894	Argillite	X	--	--	--	X	X	X	--	X	--	--
	MC4	Mixed	X	--	--	--	X	X	--	--	X	--	--
SGS (Project 12488-002)	SM1	Argillite	X	X		--	X	X	X	--	X	--	--
	SM2	Argillite	X	X	X	--	X	X	X	--	X	X	--
	SM3	Tuff	X	X	X	--	X	X	X	--	X	--	--
	SM4	Tuff	--	X	--	--	--	--	--	--	--	--	--
	SM5	Siltstone	--	X	--	--	--	--	--	--	--	--	--
	SM6	Argillite	X	X	X	--	X	X	X	--	X	--	--
	SM7	Tuff	X	X	--	--	X	X	X	--	X	--	--

Project	Sample ID	Type of Rock	Head Assays	Comminution RWI & BWI	JK DWT	E-GRG	Gravity Separa-tion	Rougher Flotation	Cleaner 2	Scv. Grav. Conc. on Clnr. Tails	Cyanidation on Clnr Conc. Std Bottle	Grav. Conc. Intensive Cyanidation	Cyanide Destruc-tion
	SM8	Tuff	X	X	--	--	X	X	X	--	X	--	--
	SM9	Argillite	X	X	X	--	X	X	X	--	X	X	--
	SM10	Argillite	X	X	X	--	X	X	X	--	X	--	--
	SM11	Crystal Tuff	X	X	--	--	X	X	X	--	X	--	--
	SM12	Siltstone	X	X	--	--	X	X	X	--	X	--	--
	SM13	Argillite	--	X	X	--	--	--	--	--	--	--	--
	SM14	Argillite	X	X	--	--	X	X	X	--	X	--	--
	SM15	Tuff	--	X	--	--	--	--	--	--	--	--	--
	SM16	Tuff	--	X	X	--	--	--	--	--	--	--	--
	SM17	Tuff	--	X	--	--	--	--	--	--	--	--	--
	SM18	Argillite	X	X	X	--	X	X	X	--	X	--	--
	SM19	Argillite	X	X	X	--	X	X	X	--	X	--	--
	SM20	Argillite	X	X	X	--	X	X	X	--	X	X	--
	SM21	Argillite	X	X	--	--	X	X	X	--	X	--	--
	SM22	Tuff	--	X	X	--	--	--	--	--	--	--	--
	SM23	Siltstone	--	X	X	--	--	--	--	--	--	--	--
	SM24	Siltstone	X	X	--	--	X	X	X	--	X	--	--
Met-Solve (Proj	865-3	Siltstone	X	--	--	--	--	X	X	X	X	--	--

Project	Sample ID	Type of Rock	Head Assays	Comminution RWI & BWI	JK DWT	E-GRG	Gravity Separa-tion	Rougher Flotation	Cleaner 2	Scv. Grav. Conc. on Clnr. Tails	Cyanidation on Clnr Conc. Std Bottle	Grav. Conc. Intensive Cyanidation	Cyanide Destruc-tion
	871B & 892	Argillite/Siltstone	X	--	--	--	--	X	X	--	--	--	--
	North Bulk		X	--	--	--	--	X	X	--	--	--	--
McClelland (Project 4373)	4373-002	Argillite/Siltstone	X	X	--	--	--	X	X	--	X	--	--
	4373-003	Argillite/Siltstone	X	X	--	--	--	X	X	--	X	--	--
	4373-004	Argillite/Siltstone	--	--	--	--	--	X	X	X	X	--	X

(*) over the course of project development the description of lithology has varied. Tuff and Graywacke are interchangeable, but Siltsone is a different rock-type.

MC4: mixed of 871 (23%w/w), 872A (15%), 872B (21%), 891 (23%), 894 (18%)

4373-004: mixed of 4373-002 and 4373-003

13.2.1 Testwork History

13.2.1.1 G&T Project KM 2637, 2010

- 3 composite samples from the main zone
- Chemical characterization
- Bulk and gravity concentrate mineralogy,
- Ore hardness BBWi
- Flowsheet development
    - Flotation (Rougher)
    - Gravity Float (Rougher)
    - Gravity Flotation (Rougher Cleaner 1) concentrate regrind & leach,
    - Primary grind size

13.2.1.2 G&T Project KM2637, 2011

- Master composite 4 sample
- Flotation optimization
- TOC removal – pre-float with fuel oil, and carboxymethyl cellulose gangue depression in cleaner circuit
- Effect of concentrate regrind, & pre-aeration
- Performance of major lithologies using selected flowsheet.

13.2.1.3 G&T KM3185, 2012

- 24 variability samples from 5 drillholes (same samples as SGS 2010 & 2012)
- Chemical characterization
- Ore hardness BBWi
- Gravity/flotation/CIL
- Flotation tailings mineralogy
- Flowsheet: Gravity, pre-float, flotation, gravity concentrate leach, float concentrate CIL
- Primary grind size 180 micron
- Effect of pre-float
- Knelson concentrate regrind size & preg-robbing,
- Effect of pre-aeration
- 2 lithology samples
- Settling and vacuum filtration rates (mill feed)

13.2.1.4 SGS Project No. 12488-01, 2010

- 24 samples from 5 drillholes
- Comminution testing (JK DWT, SMC, Bond RWi, Bond BWi, Abrasion)

13.2.1.5 SGS Project No. 12488-003, 2011

- 2 samples
- HPGR amenability testing

13.2.1.6 Knelson Project KRTC 20559, 2010

- 2 samples from the main zones
- E-GRG test

13.2.1.7 Knelson Project KRTC 20559-1, 2011

- Gravity modeling

13.2.1.8 SGS Project 12488-002 – Report 1, 2012

- Gravity, deslime/pre-float, flotation flowsheet
- Falcon vs Knelson test 10kg
- Deslime/pre-float at 90 vs 184 µm

13.2.1.9 SGS Project 12488-002 – Report 2, 2012

- 4 variability samples from 5 drillholes
- Gravity, deslime, flotation (Rougher, 3 Cleaners)
- Gravity + float concentrate regrind CIL tests (35% solids, O2 pre-aeration, 2- 10g/l NaCN)
    - 11-22 µm, 24h gravity concentrate
    - 10-110 µm 48h float concentrate

13.2.1.10 Met-Solve Project MS1735, 2017

- 6 composite samples, incl. North zone
- Flowsheet simplification: Flotation (Ro, Cl, Re-Cl), regrind, concentrate CIL (180 µm primary grind)
- Cleaner/recleaner tailings gravity scavenger
- CIL at 48h, 20 micron, 2g/l NaCN, 12h pre-aeration

13.2.1.11 McClelland MLI Job No. 4373, 2019

- 4 composite samples, included low grade,
- Flowsheet – flotation, gravity scavenger on cleaner tailings, flotation conc regrind, CIL (180 µm primary grind)
    - Flotation kinetic tests
    - Concentrate regrind size,
    - CIL & Cyanide detox tests
    - Comminution tests – CWi, Ai, BBWi
    - Scavenger gravity on rougher tailings

13.3 Sample Origin

The location of the drillholes for the different metallurgical testwork programs is presented in Figure 13-1 and Figure 13-2. These testwork programs provide good coverage of the planned mine area and are representative of the anticipated ore feed grade. This includes the main lithologies such as Argillite, Siltstone, and Tuff. Most of the metallurgical testwork was performed from samples which originate from the central part of the open pit (called the main zone).

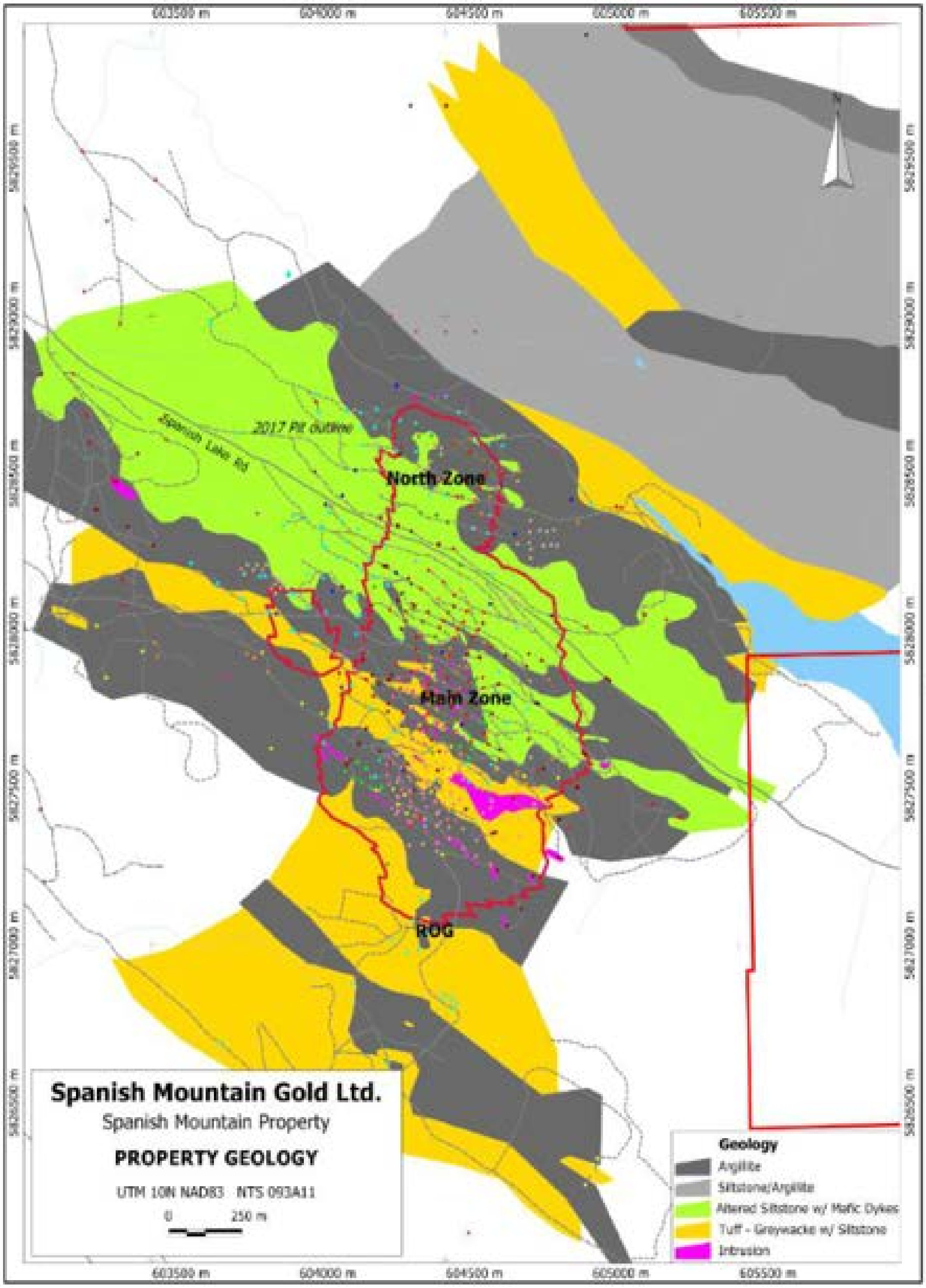


Figure 13-1 Approximate Open Pit Outline Showing the Main Zone

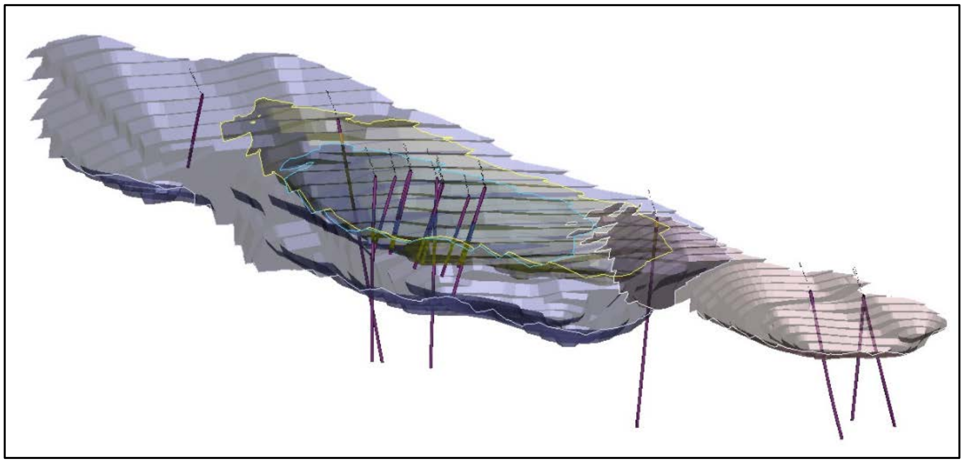


Figure 13-2 Cross-section of the Open Pit Showing the Drillholes for Metallurgical Testwork

13.4 Testwork Results

13.4.1 Head Analysis

Head grades of various test samples have been characterized by assay and the results have been detailed in several reports. Selected head assays are presented in Table 13-3 and Table 13-4.

Table 13-3 Summary of Head Assays

Project	Sample ID	Type of Rock	Head Assays						
			Au (g/t)	Ag (g/t)	Fe (%)	S_{tot} (%)	Sulphur (S^{2-}) (%)	TOC (%)	TIC (%)
G&T (Project KM2637)	865-1	Siltstone*	0.45	1.2	4.8	1.3	1.3	0.28	3.03
	865-2	Argillite	0.94	1.2	4.1	2.9	2.9	1.18	2.04
	865-3	Siltstone*	0.82	1.3	3.3	1.5	1.4	0.26	2.05
	871	Argillite/Siltstone	1.05	1.4	4.3	3.3	3.4	1.09	1.84
	871B	Siltstone*	0.36	0.9	3.2	1.5	1.3	0.65	1.25
	872A	Argillite	0.54	3.2	3.9	2.9	2.8	1.01	1.98
	872B	Argillite/Siltstone	0.99	1.3	4.2	3.0	2.9	0.24	2.01
	891	Siltstone/Argillite	1.33	1.1	3.9	2.7	2.7	1.14	1.48
	894	Argillite	1.26	1.1	4.0	2.3	2.3	0.70	2.01
	MC4	Mixed	1.02	1.2	4.0	2.6	2.7	0.78	1.94
SGS (Project 12488-002)	SM1	Argillite	3.25	0.8	3.6	1.0	--	0.74	--
	SM2	Argillite	1.08	0.9	4.8	2.9	--	1.30	--
	SM3	Tuff	0.32	0.7	4.5	1.0	--	0.50	--
	SM6	Argillite	2.42	4.4	3.9	2.4	--	1.12	--
	SM7	Tuff	0.20	0.5	4.6	0.7	--	0.48	--
	SM8	Tuff	0.21	0.5	5.2	0.5	--	0.55	--
	SM9	Argillite	0.36	1.0	3.1	1.1	--	1.36	--
	SM10	Argillite	0.24	0.7	3.6	2.1	--	1.56	--
	SM11	Crystal Tuff	0.15	0.9	3.8	0.4	--	0.60	--
	SM12	Siltstone	0.36	0.8	4.9	2.1	--	0.82	--
	SM14	Argillite	0.94	0.6	4.3	3.2	--	0.93	--
	SM18	Argillite	0.34	1.5	4.7	4.1	--	1.50	--

Project	Sample ID	Type of Rock	Head Assays						
			Au (g/t)	Ag (g/t)	Fe (%)	S_{tot} (%)	Sulphur (S^{2-}) (%)	TOC (%)	TIC (%)
	SM19	Argillite	1.10	1.2	4.5	3.6	--	1.56	--
	SM20	Argillite	1.18	1.1	4.3	3.7	--	1.90	--
	SM21	Argillite	0.29	0.6	2.6	0.5	--	0.96	--
	SM24	Siltstone	0.53	1.2	2.9	1.8	--	1.53	--
Met-Solve (Project MS1735)	865-3	Siltstone	1.06	--	4.1	1.7	--	0.37	--
	871B & 891	Argillite/Siltstone	1.15	--	3.9	2.1	--	1.35	--
	North Bulk	--	0.91	--	5.0	2.5	--	1.71	--
McClelland (Project 4373)	4373-001	Argillite/Siltstone	0.36	0.5	4.7	1.5	1.2	0.58	2.21
	4373-002	Argillite/Siltstone	0.82	0.5	4.1	1.9	1.7	0.80	1.85
	4373-003	Argillite/Siltstone	1.02	1.2	3.6	1.9	1.8	0.60	1.88
	4373-004	Argillite/Siltstone	0.92	0.87	3.87	1.89	1.74	0.70	1.86

(*) over the course of project development the description of lithology has varied. Tuff and Graywacke are interchangeable, but Siltsone is a different rock-type.
MC4: mixed of 871 (23%w/w), 872A (15%), 872B (21%), 891 (23%), 894 (18%).
4373-004: mixed of 4373-002 and 4373-003

Table 13-4 Summary of Head Chemical Characterization

Element	Units	SGS (Project 12488-002)																McClelland (Project 4373)		
		SM1	SM2	SM3	SM6	SM7	SM8	SM9	SM10	SM11	SM12	SM14	SM18	SM19	SM20	SM21	SM24	4373-001	4373-002	4373-003
Au	g/t	3.25	1.08	0.32	2.42	0.20	0.21	0.36	0.24	0.15	0.36	0.94	0.34	1.10	1.18	0.29	0.53	0.36	0.82	1.02
Ag	g/t	0.8	0.9	0.7	4.4	0.5	0.5	1.0	0.7	0.9	0.8	0.6	1.5	1.2	1.1	0.6	1.2	0.5	0.5	1.2
Fe	%	3.6	4.8	4.5	3.9	4.6	5.2	3.1	3.6	3.8	4.9	4.3	4.7	4.5	4.3	2.6	2.9	4.7	4.1	3.6
S	%	1.0	2.9	1.0	2.4	0.7	0.5	1.1	2.1	0.4	2.1	3.2	4.1	3.6	3.7	0.5	1.8	1.6	1.9	2.0
TOC	%	0.74	1.30	0.50	1.12	0.48	0.55	1.36	1.56	0.60	0.82	0.93	1.50	1.56	1.90	0.96	1.53	0.58	0.80	0.60
As	g/t	<40	<40	<40	<40	<40	<40	<40	<40	<40	<40	<40	<40	<40	<40	<40	<40	79	104	99
Co	g/t	19	20	22	15	17	22	17	17	13	24	22	25	23	21	16	17	17	15	13
Cu	g/t	77	110	95	95	60	96	97	100	79	86	85	64	110	80	67	81	88	83	62
Ni	g/t	28	44	<20	40	<20	22	110	89	<20	36	35	110	91	90	74	77	22	31	20
Pb	g/t	<50	<50	<50	<50	<50	<50	<50	<50	130	<50	86	<50	220	<50	<50	<50	15	20	35
Sb	g/t	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10	1.6	0.6	0.6
Zn	g/t	120	180	73	240	78	56	240	220	140	150	310	250	700	260	160	200	167	176	137
Hg	g/t	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	0.013	0.017	0.011
Mn	g/t	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	1080	797	795

These results conclude that total organic carbon is present in high enough concentrations to warrant removal and thus minimise loss in gold leach recovery because of preg-robbing, this has been incorporated in the flotation process. No other elements are present at levels that are cause for concern.

13.4.2 Mineralogy Analysis

Gold mineralization has been described in historical technical reports as follows:

SMG deposit is a gold-based sediment-hosted vein deposit. Gold occurs as free gold associated with quartz veins and as attachments to and occlusions in pyrite.

Gold mineralization occurs as two main types:

- Disseminated within the black, graphitic argillite. This is the most economically significant form. Gold grain size is typically less than 30 microns, and is often, but not always, associated with pyrite. Disseminated gold has also been associated with quartz veins in the argillite.
- Within quartz veins in the siltstone/tuff/greywacke sequences. It occurs as free, fine to coarse (visible) gold and can also be associated with sulphides including galena, chalcopyrite, and sphalerite.

A petrographic study performed in 2006 showed variable carbonaceous siltstone/mudstones with fine grained greywackes. In some instances, up to 30% of the mineralization was carbonaceous material. Native gold was identified in four samples as inclusions and fracture- fill in pyrite on crystal boundaries between pyrite crystals and in the gangue adjacent to pyrite. The particles were very fine-grained, less than 20 µm and generally less than 5 µm and were described as occurring in 15 of the 21 samples studied. There was no clear indication from the study whether the gold was preferentially associated with any habit of pyrite, or other mineral type.

13.4.3 Comminution Test Results

Comminution testwork on samples from different lithologies conducted by SGS is summarized in Table 13-5.

Table 13-5 Comminution Testwork Results

Argillite								Siltstone								Tuff							
Sample ID	S.G.	Ai [g]	Cwi [kWh/t]	Rwi [kWh/t]	Bwi [kWh/t]	SMC [A*b]	JK DWT [A*b]	Sample ID	S.G.	Ai [g]	Cwi [kWh/t]	Rwi [kWh/t]	Bwi [kWh/t]	SMC [A*b]	JK DWT [A*b]	Sample ID	S.G.	Ai [g]	Cwi [kWh/t]	Rwi [kWh/t]	Bwi [kWh/t]	SMC [A*b]	JK DWT [A*b]
SM-1	2.66	0.22	9.1	12.4	10.9	54.0	--	SM-5	2.77	0.26	11.5	16.1	14.7	39.8	--	SM-3	2.74	0.21	14.4	14.1	12.9	42.6	46.7
SM-2	2.73	0.11	11.4	13.2	11.9	51.4	52.5	SM-12	2.73	0.27	14.6	16.7	15.7	31.8	--	SM-4	2.77	0.25	14.7	15.1	13.3	32.8	--
SM-6	2.82	0.23	12.0	13.8	12.5	52.0	47.9	SM-23	2.70	0.28	12.8	14.1	15.9	51.2	50.1	SM-7	2.78	0.20	14.9	15.3	13.5	35.7	--
SM-9	2.74	0.22	10.5	14.4	13.3	52.0	53.9	SM-24	2.74	0.26	11.6	14.2	15.3	51.7	--	SM-8	2.75	0.22	13.4	15.1	13.3	33.0	--
SM-10	2.77	0.30	11.8	14.8	13.8	45.1	47.8	--	--	--	--	--	--	--	--	SM-11	2.74	0.28	18.0	17.5	16.7	27.0	--
SM-13	2.82	0.20	8.9	13.3	13.0	49.5	46.6	--	--	--	--	--	--	--	--	SM-15	2.79	0.19	11.6	17.0	16.0	26.0	--
SM-14	2.81	0.17	11.1	12.9	13.3	49.8	--	--	--	--	--	--	--	--	--	SM-16	2.80	0.21	17.0	16.3	15.1	27.0	28.7
SM-18	2.83	0.28	9.9	12.7	12.6	47.9	45.8	--	--	--	--	--	--	--	--	SM-17	2.80	0.30	15.2	16.0	14.5	26.0	--
SM-19	2.78	0.28	10.6	12.4	12.7	83.7	83.2	--	--	--	--	--	--	--	--	SM-22	2.82	0.12	12.4	13.9	12.1	37.4	36.4
SM-20	2.79	0.30	12.9	13.8	13.7	50.7	54.7	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
SM-21	2.63	0.21	11.4	13.8	12.9	38.4	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Average	2.76	0.23	10.9	13.4	12.8	52.2	54.1	--	2.74	0.27	12.6	15.3	15.4	43.6	50.1	--	2.78	0.22	14.6	15.6	14.2	31.9	37.3
75th	2.82	0.28	11.6	13.8	13.3	52.0	54.1	--	2.75	0.27	13.3	16.3	15.8	51.3	50.1	--	2.80	0.25	15.2	16.3	15.1	35.7	41.6
Max	2.83	0.30	12.9	14.8	13.8	83.7	83.2	--	2.77	0.28	14.6	16.7	15.9	51.7	50.1	--	2.82	0.30	18.0	17.5	16.7	42.6	46.7
Min	2.63	0.11	8.9	12.4	10.9	38.4	45.8	--	2.70	0.26	11.5	14.1	14.7	31.8	50.1	--	2.74	0.12	11.6	13.9	12.1	26.0	28.7

The comminution testwork results indicate that the Spanish Mountain ore is highly variable in competency and can be classified as competent and moderately hard, and moderately abrasive compared to the JKTech database.

13.4.4 Flotation

This section summarizes the results of the relevant flotation test programs.

13.4.4.1 G&T Testwork (Project KM2637)

Testwork was conducted on three composite samples derived from drillhole DDH-865. Overall rougher flotation test results for the three composites in terms of gold recovery are summarized in Table 13-6.

Table 13-6 Rougher Flotation Results (average)

Composite	Type of Rock	Au Feed Grade (g/t)	Feed S (%)	Au Recovery (%)	Rougher Wt. (%)
865-1	Siltstone	0.45	1.4	97.9	9.9
865-2	Argillite	0.94	3.0	96.6	16.1
865-3	Siltstone	0.82	1.4	96.5	11.1

Both the primary grind size evaluation and the sensitivity of rougher flotation tailings to grind size were evaluated based on a single drillhole sample (865-3). See Figure 13-3 and Figure 13-4.

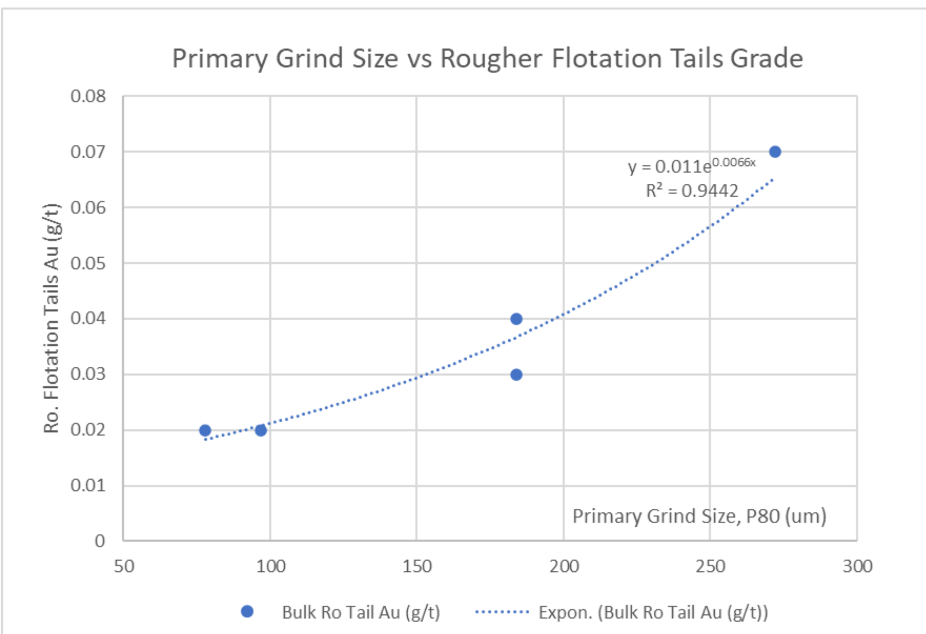


Figure 13-3 Primary grind size vs rougher flotation tailings grade

Overall recovery (gravity /rougher flotation) as a function of flotation time at four grind sizes is summarized in Figure 13-4.

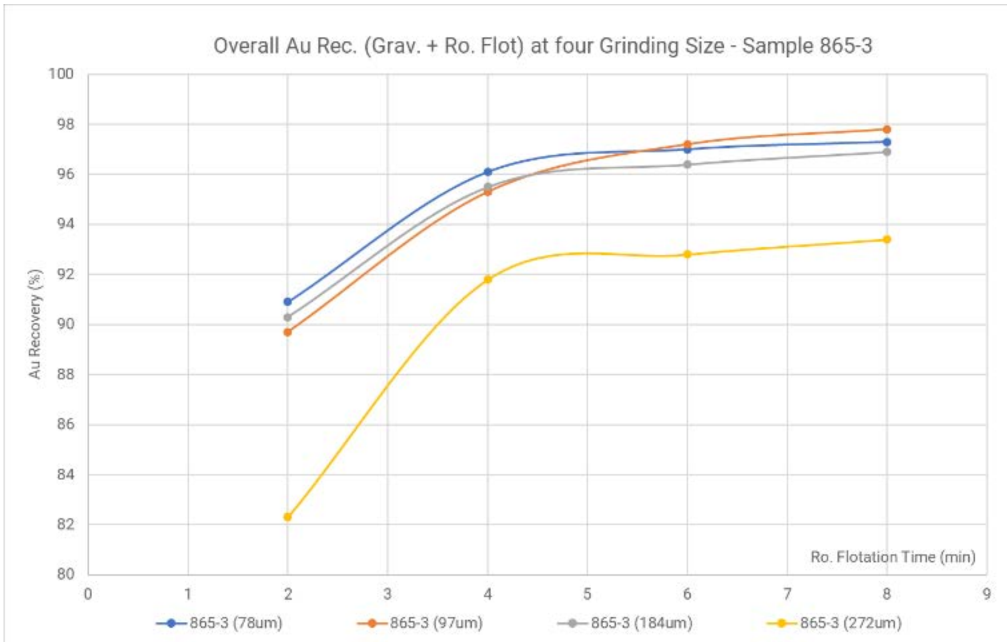


Figure 13-4 Sensitivity of Rougher Flotation to Grind Size

Kinetic data for two composites, rougher and cleaner stages is shown in Figure 13-5 through Figure 13-8.

The summary of flotation results, without gravity upstream of flotation for Argillite, Siltstone and Mixed is presented in Table 13-7 through Table 13-9.

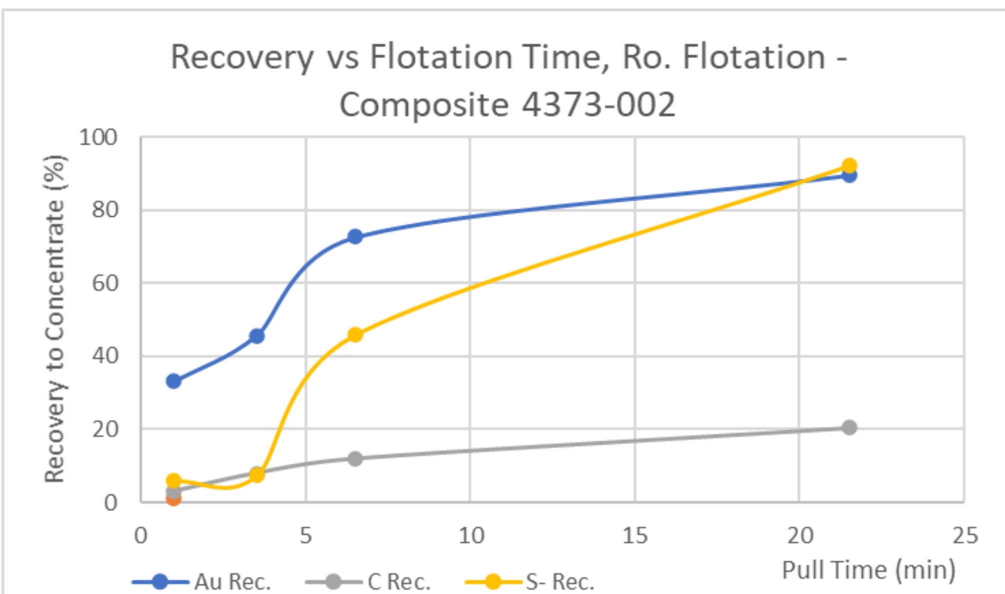


Figure 13-5 Kinetic Rougher Composite 4373-002, 80%- 180µm Feed Size

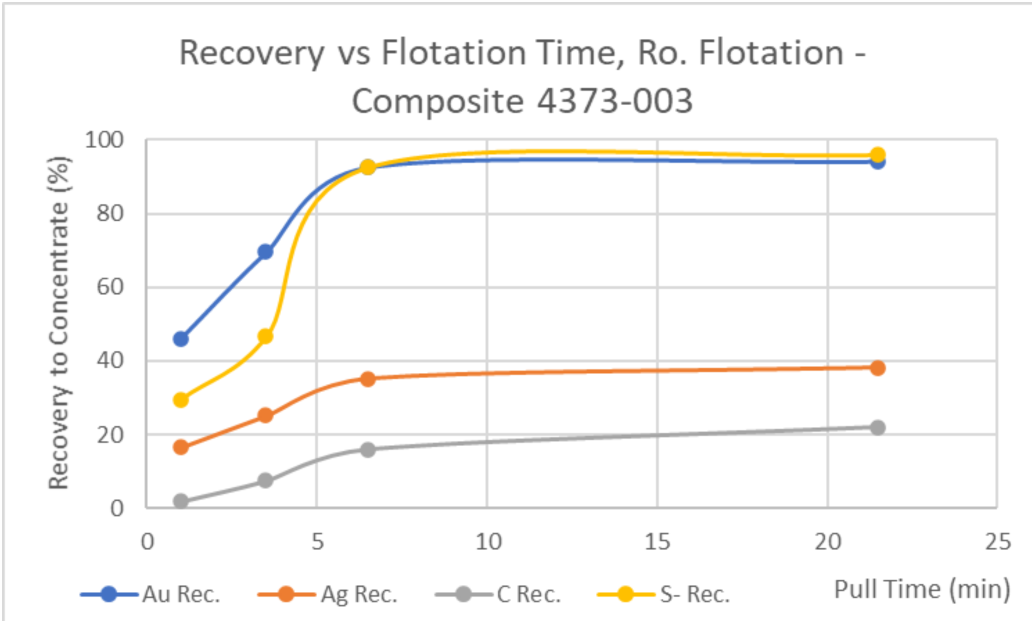


Figure 13-6 Kinetic Rougher Flotation Composite 4373-003, 80%-180µm Feed Size

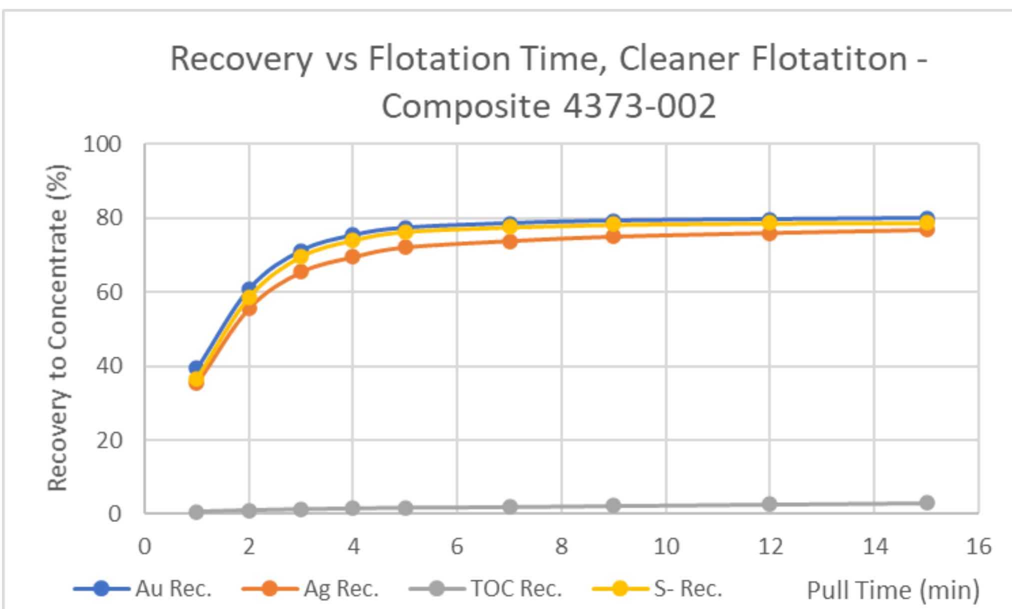


Figure 13-7 Kinetic Cleaner Flotation Composite 4373-002, 80%-180µm Feed Size

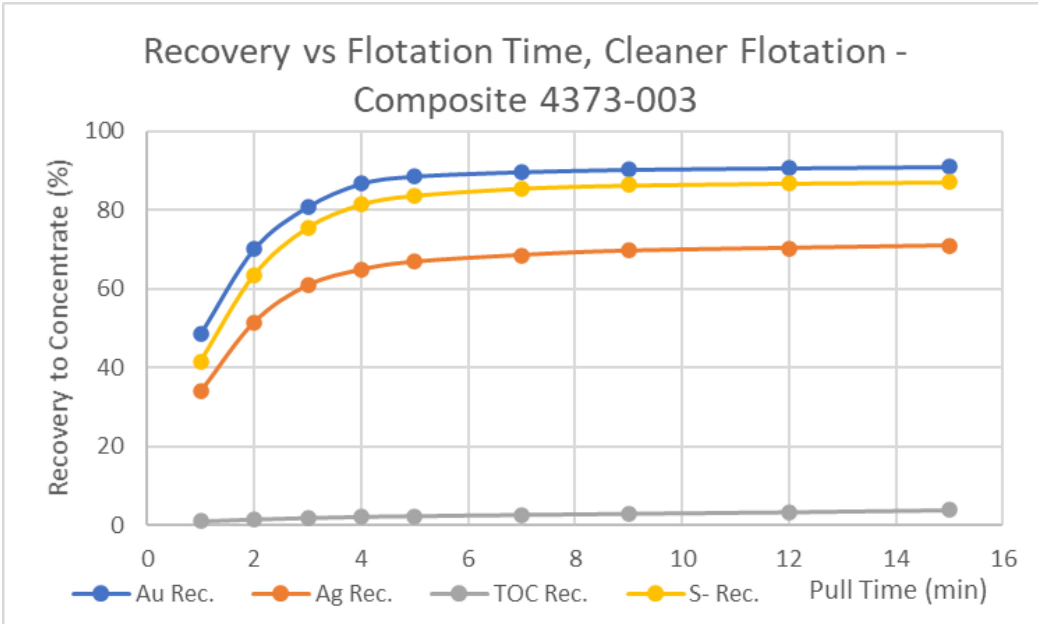


Figure 13-8 Kinetic Cleaner Flotation Composite 4373-003, 80%-180µm Feed Size

Table 13-7 Summary of Flotation Results, without gravity upstream of flotation for Argillite, Siltstone and Mixed

Test No.	Sample ID	Grind size K80 (um)	Overall Mass Pull (%)	Head Assay Au (g/t)	1st Cl. Conc. Au (g/t)	1st Cl. Conc. Au Rec. (%)	Ro. Tail Au (g/t)	Head Assay Ag (g/t)	1st Cl. Conc. Ag (g/t)	1st Cl. Conc. Ag Rec. (%)	Ro. Tail Ag (g/t)	1st Cl. Conc. TOC grade (%)
Argillite												
KM2637-08[a]	865-2	189	16.2	1.01	6.0	96.6	0.04	1.0	7.0	85.0	0.3	--
Siltstone												
KM2637-05[a]	865-3	184	8.9	0.89	9.6	96.2	0.04	1.0	7.0	74.5	0.2	--
KM2637-06[a]	865-1	180	7.4	0.47	6.1	96.0	0.02	1.0	5.0	33.2	1.0	--
KM2637-10[b]	865-3	184	6.9	0.76	10.3	93.8	0.05	1.0	8.0	44.0	1.0	--
Argillite/Siltstone												
KM2637-56[d]	891	192	7.3	1.10	13.8	92.0	0.07	1.0	6.2	43.1	0.6	1.1
KM2637-64[c]	MC4	200	8.1	0.87	9.9	91.5	0.05	1.6	10.8	55.6	0.7	1.1

a. Only Ro. Flotation 4 stages
b. Ro. Flotation (3 stages) + one single Cleaner
c. Pre-float + Ro. Flotation (3 stages) + one single Cleaner
d. Pre-float 2 + Ro Flotation (3 Stages) + one single Cleaner

Table 13-8 Summary of Flotation Results, with gravity upstream of flotation for Argillite, Siltstone and Mixed

Test No.	Sample ID	Grind size K80 (um)	Overall Mass Pull (%)	Head Assay Au (g/t)	1st Cl. Conc. Au (g/t)	1st Cl. Conc. Au Rec. (%)	1st Cl. Conc. TOC grade (%)	Ro. Tail Au (g/t)	Overall (Grav./Flot.)Au Rec. (%)	Head Assay Ag (g/t)	1st Cl. Conc. Ag (g/t)	1st Cl. Conc. Ag Rec. (%)	Ro. Tail Ag (g/t)	Overall (Grav./Flot.) Ag Rec. (%)
Argillite														
KM2637-25	865-2	189	18.6	1.07	4.5	77.7	--	0.04	96.6	1.0	3.0	51.5	0.0	62.7
Siltstone														
KM2637-15	865-3	184	**6.1**	1.21	7.3	37.1	--	0.01	98.3	1.0	6.0	36.9	0.0	50.7
Mixed														
KM2637-35	MC4	173	6.9	0.90	7.5	57.5	2.6	0.06	85.0	1.1	6.0	36.4	0.6	46.9

SGS Testwork (Project 12488-002)

Table 13-9 Summary of Flotation Results, with gravity and deslime ahead of flotation for Argillite, Siltstone and Tuff

Test No.	Sample ID	Grind size K80 (um)	Overall Mass Pull (%)	Head Assay Au (g/t)	3rd Cl. Conc. Au (g/t)	3rd Cl. Conc. Au Rec. (%)	3rd Cl. Conc. TOC grade (%)	Ro. Tail Au (g/t)	Overall (Grav./Flot.) Au Rec. (%)	Head Assay Ag (g/t)	3rd Cl. Conc. Ag (g/t)	3rd Cl. Conc. Ag Rec. (%)	Ro. Tail Ag (g/t)	Overall (Grav./Flot.) Ag Rec. (%)
Argillite														
SM1-1	SM1	188	1.4	0.40	12.5	37.3	0.2	0.02	85.0	0.50	11.9	22.3	0.5	30.6
SM2-1	SM2	187	3.2	1.08	14.6	51.7	0.4	0.12	80.5	0.90	12.2	37.5	0.5	45.4
SM6-1	SM6	193	2.5	1.14	23.5	42.9	0.2	0.12	80.0	0.80	12.3	24.8	0.5	55.8
SM9-1	SM9	171	1.4	0.36	10.6	38.4	0.6	0.03	92.1	1.00	19.1	27.9	0.7	41.2
SM10-1	SM10	194	1.4	0.24	8.8	37.9	0.4	0.08	72.0	0.70	12.0	21.6	0.5	28.7
SM14-1	SM14	171	4.1	0.94	10.1	44.4	0.5	0.14	86.7	0.60	6.0	30.6	0.5	40.3
SM18-1	SM18	165	1.4	0.34	7.6	25.8	0.4	0.10	49.2	1.50	9.0	12.1	0.5	15.7
SM19-1	SM19	187	3.9	1.10	19.8	40.7	0.5	0.11	85.7	1.20	14.9	38.0	0.5	55.8
SM20-1	SM20	172	1.9	1.18	14.5	34.5	0.6	0.10	81.7	1.10	9.7	19.1	0.5	29.3
SM21-1	SM21	176	0.6	0.29	11.6	24.0	0.9	0.14	58.7	0.60	23.5	20.4	0.5	24.3
Siltstone														
SM12-1	SM12	194	1.5	0.36	10.1	46.1	0.5	0.07	78.0	0.80	10.6	22.2	0.5	28.4
SM24-1	SM24	182	1.1	0.53	13.7	34.9	0.7	0.12	69.6	1.20	15.8	19.0	0.5	25.3
Tuff														
SM3-1	SM3	170	1.5	0.32	8.2	43.3	0.4	0.02	87.6	0.70	15.3	29.1	0.5	33.6
SM7-1	SM7	166	1.7	0.20	9.9	54.5	0.4	0.02	90.6	0.50	13.7	29.4	0.5	36.3
SM8-1	SM8	166	0.6	0.21	11.6	28.5	0.6	0.03	84.1	0.50	21.7	17.7	0.5	26.3
SM11-1	SM11	193	0.1	0.15	9.7	13.8	0.3	0.04	59.9	0.90	30.1	6.4	0.5	21.7

13.4.4.2 G&T Testwork (Project KM3185)

Variability testing on the same samples used by SGS was carried out in parallel at G&T. The flowsheet difference from SGS program 12488-002 was that a pre-float rather than desliming step was incorporated ahead of flotation.

Table 13-10 Summary Flotation Results (2nd Cl. Conc.), with upstream gravity and pre-float

Test No.	Sample ID	Lithology	Head Assay Au (g/t)	Head Assay Ag (g/t)	Grind size K80 (um)	2nd Cl. Conc. Au (g/t)	2nd Cl. Conc. Ag (g/t)	2nd Cl. Conc. Mass (%)	Ro. Tail Au (g/t)	Ro. Tail Ag (g/t)
KM3185-10	SM-9	Argillite	0.58	1	323	13.7	21	1.4	0.06	1.0
KM3185-11	SM-10	Argillite	0.33	1	194	5.81	10	2.7	0.04	1.0
KM3185-12	SM-18	Argillite	0.52	1	145	4.6	5	4	0.11	1.0
KM3185-13	SM-19	Argillite	1.93	2	171	24.4	13	5.8	0.08	1.0
KM3185-14	SM-3	Tuff	0.32	1	169	6.92	12	1.4	0.02	1.0
KM3185-15	SM-6	Argillite	1.22	2	59	18.4	11	3.9	0.04	1.0
KM3185-16	SM-7	Tuff	0.28	1	148	10.4	12	0.9	0.02	1.0
KM3185-17	SM-8	Tuff	0.38	1	181	10.3	14	0.6	0.01	1.0
KM3185-18	SM-20	Argillite	1.25	2	139	13.6	8	5	0.06	1.0
KM3185-20	Mixed Comp 25	Mixed Comp 25	0.3	1	162	5.3	8	1.4	0.02	1.0
KM3185-27	SM-18	Argillite	0.4	2.0	134	4.52	4.0	4.2	0.05	1.0
KM3185-28	SM-9	Argillite	0.42	5	183	14.9	2	2.4	0.03	4.0
KM3185-29	928A	Comp 928A	0.44	1	147	6.62	7	4.2	0.04	1.0
KM3185-30	928C	Comp 928C	0.37	1	175	6.16	6	3.7	0.03	1.0
KM3185-31	928D	Comp 928D	0.78	1	141	14	8	3.9	0.03	1.0
KM3185-32	928D	Comp 928D	0.44	1	144	10.5	16	2.2	0.03	1.0
KM3185-33	939A	Comp 939A	0.73	2	134	12.3	9	2.1	0.1	1.0
KM3185-34	939B	Comp 939B	0.26	2	131	2.62	8	2.7	0.15	1.0
KM3185-35	981	Comp 981	0.88	1	125	22.4	14	1.6	0.04	1.0

Note: Test 27 Rougher PAX addition was 150g/t, test 33 &34 PAX addition was 120g/t, all other tests were 90g/t

13.4.4.3 Met-Solve Testwork (Project MS1735)

Table 13-11 Summary Flotation Results (2nd Cl. Conc.), without gravity upstream for Siltstone and Siltstone/ Argillite

Test No	Sample ID	Grind size K80 (um)	Overall Mass Pull (%)	Head[1] Calc Au (g/t)	2nd Cl. Conc. Au (g/t)	2nd Cl. Conc. Au Rec. (%)	Ro. Tail Au (g/t)	2nd Cl. Conc. TOC grade (%)
Siltstone								
ND201	865-3	185	2.4	1.06	38.8	86.8	0.09	0.17
ND211	865-3	185	2.9	1.00	30.2	86.5	0.09	0.21
Siltstone/Argillite								
ND101B	871B & 891	179	3.3	1.10	27.9	82.5	0.11	0.28
ND102	871B & 891	171	2.5	0.95	32.0	84.8	0.12	0.17
ND103	871B & 891	174	2.7	1.13	35.7	85.1	0.12	0.26
ND104	871B & 891	176	2.2	2.84	115.4	90.6	0.17	0.36
ND105	871B & 891	174	2.4	0.96	32.9	82.2	0.12	0.28
ND106	871B & 891	162	2.9	1.38	38.9	81.4	0.16	0.23
ND107	871B & 891	178	2.6	1.46	47.8	85.1	0.13	0.31

Note: Head Au assay for Siltstone/Argillite: 1.15 g/t and for Siltstone: 1.06 g/t

13.4.4.4 McClelland Testwork (Project MLI4373)

The 2019 McClelland testwork is the most recent. The potential impact of gravity concentration and the reasons for incorporating this unit operation as part of the process flowsheet ahead of flotation is discussed in Section 13.4.6.2.1.

Summary results from scoping flotation test and bulk flotation/gravity concentration test are shown in Table 13-12 and Table 13-13.

Table 13-12 Summary Results, Flotation Concentration without gravity upstream, Composites, 80%-180 µm Feed Size

Test No.	Sample ID	Grind size K80 (um)	Overall Mass Pull (%)	Head[1] Calc Au (g/t)	2nd Cl. Conc. Au (g/t)	2nd Cl. Conc. Au Rec. (%)	2nd Cl. Conc. Sulfide S- grade (%)	Ro. Tail Au (g/t)	Head[1] Calc Ag (g/t)	2nd Cl. Conc. Ag (g/t)	2nd Cl. Conc. Ag Rec. (%)	Ro. Tail Ag (g/t)
Siltstone/Argillite												
4373-002	F-1	180	3.2	0.90	25.5	91.3	32.2	0.07	1.0	14	31.7	<1
4373-002	F-4	180	3.2	1.18	34.5	92.0	35.3	0.08	1.0	14	31.1	<1
4373-003	F-2	180	3.4	0.95	24.4	86.0	34.6	0.07	1.0	15	34.3	<1
4373-003	F-5	180	3.4	0.92	24.5	90.5	33.1	0.06	1.0	14	32.7	<1

Rougher/cleaner/recleaner flotation tests were conducted on each of the three individual composites to confirm response to flotation treatment using conditions optimized during previous testing programs with different laboratories.

Scoping rougher/cleaner/recleaner flotation test results showed that Composites 4373-002 and 4373-003 responded well to flotation treatment, at an 80%-180 um feed size. The flotation recleaner concentrate produced was an average of 3.3% of the feed weight, had an average grade of 27.2 g/t Au, and represented an average gold recovery of 90%. Gold recovery from the lower grade composite (4373-001) was 79.1%. This was the only test conducted on this composite.

Results from the bulk test conducted on master composite 4373-004 showed that a final concentrate produced was 3.06% of the feed weight, assayed 27.9 g/t Au, and represented a gold recovery of 88.2%.

Table 13-13 Combined Bulk Flotation/Scv. Gravity Concentration, Composite 4373-004, 80%-180 um Feed Size

Composite	Final Conc. Weight (%)	Final Conc. Au (g/t)	Final Conc. Ag (g/t)	Ro. Tails Au (g/t)	Calc'd. Head Au (g/t)	Calc'd. Head Ag (g/t)	Final Conc. Au Rec. (%)	Final Conc. Ag Rec. (%)
4373-004	3.06	27.9	15	0.08	0.97	1.0	88.2	30.7

4373-004: mixed of 4373-002 (50%) and 4373-003 (50%)
4373-004: rougher/cleaner/recleaner flotation followed by scavenger flotation and gravity concentration of the scavenger flotation cleaner tailings.

13.4.5 Gravity Concentration

This section summarizes the results of the relevant test programs investigating gravity recovery ahead of flotation. Table 13-14 present the average values of gold recovery from project KM2637 (G&T testwork) and project 12488-002 (SGS testwork).

Table 13-14 Summary of Results, Average Gold Recovery - Argillite, Siltstone, Mixed (Argillite/Siltstone) and Tuff material

Project No.	Type of Rock	Grind size K80 (um)	Mass Pull (%)	Head Assay Au (g/t)	Grav. Conc. Au (g/t)	Grav. Conc. Au Rec. (%)	Grav. Tail Au (g/t)	Head Assay Ag (g/t)	Grav. Conc. Ag (g/t)	Grav. Conc. Ag Rec. (%)	Grav. Tail Ag (g/t)	Grav. Conc. TOC grade (%)
12488-002	Argillite	180	0.9	0.71	33.6	39.4	0.5	0.9	13	11.3	0.9	1.0
KM2637	Argillite	178	1.3	0.78	19.5	32.6	--	2.5	41	19.9	--	0.4
12488-002	Siltstone	188	1.0	0.45	13.5	33.3	0.3	1.0	5	6.3	0.8	0.8
KM2637	Siltstone	173	1.0	0.83	36.0	44.9	--	1.0	15	14.3	--	0.2
KM2637	Mixed	189	1.4	1.00	20.8	27.5	--	1.3	16	15.8	--	0.4
12488-002	Tuff	174	0.9	0.22	12.0	45.5	0.1	0.7	7	8.8	0.7	0.5

The mass pull to gravity was 0.9-1.4% which is higher than what would be expected for plant operation (typically less than 0.05%).

13.4.5.1 Knelson E-GRG Gravity Recovery and Modelling (Project KRTC 20559)

Extended Gravity Recoverable Gold (EGRG) testing was carried out by the Knelson Research and Technology Centre at three grind sizes for two samples. The overall GRG for sample 865-2 after three stages of grinding was 57.7%. The recoveries from stage 1, 2 and 3 were 15.6%, 20.6% and 21.6% respectively with a final grind P80 of 53 microns.

The P20, P50 and P80 values for sample 865-2 were 25, 53, and 120 microns, respectively. Based on these values, the grain size of the GRG in the sample is classified as Fine to Moderate when compared to the gold grain size classification scale based on Amira's project P420B.

The overall gravity recovery from the EGRG test was 65.2% for sample 865-3. The recoveries from stage 1, 2 and 3 were 26.1%, 22.5% and 16.6% and the P20, P50 and P80 values for sample 865-3 were 25, 75, and 300 microns, respectively. From these values the grain size of the GRG in the sample provided is classified as Coarse to Very Coarse.

The EGRG test results were used by Knelson to model the expected plant performance, as shown in Table 13-15.

Table 13-15 E-GRG Gravity Modelling Results for Argillite (866-2) and Siltstone (865-3) samples

Equipment	Gravity Feed Rate (mtph)	Circ. Load Treated (%)	Gravity Rec. 175um grind.	
			865-2 (% Au)	865-3 (% Au)
1-KC-QS48	300	8	7.9	10.4
2-KC-QS48	600	16	11.8	14.9
3-KC-QS48	900	24	14.4	17.8
4-KC-QS48	1200	32	16.4	20.0
5-KC-QS48	1500	40	18.1	21.7
6-KC-QS48	1800	48	19.5	23.1
7-KC-QS48	2100	56	20.7	24.2
8-KC-QS48	2400	64	21.8	25.3
9-KC-QS48	2700	72	22.8	26.2
10-KC-QS48	3000	80	23.8	27.0

13.4.6 Scavenger Gravity Concentration[1]

13.4.6.1 Met-Solve Testwork (Project MS1735)

The cleaner and re-cleaner flotation tailings from a bulk flotation test were combined and subjected to a low mass yield, high G-force gravity test using a Falcon L40 concentrator. The results are presented in Table 13-16.

Table 13-16 Summary of Scavenger Gravity Concentration - Siltstone

Test No.	Sample ID	Grind size K80 (um)	Mass Pull (%)	Head Assay Au (g/t)	Grav. Conc. Au (g/t)	Grav. Conc. Au Rec. (%)	Grav. Tail Au (g/t)	Head Assay TOC (%)	Grav. Conc. TOC (%)	Grav. Conc. TOC Rec. (%)	Grav. Tail TOC (%)
ND231	865-3	185	8.6	0.47	8.23	76.7	0.24	0.9	0.5	4.8	0.89

13.4.6.2 McClelland Testwork (Project MLI4373)

Gravity concentration tests were conducted on the combined 1st and 2nd cleaner tailings produced during scavenger flotation of the products from bulk flotation tests on composite 4373-004. These tests were conducted to simulate the planned process flowsheet, which includes gravity concentration processing of the flotation cleaner tailings. Initially, a gravity concentration test (G-1/G-2) was conducted on the scavenger flotation tailings to evaluate the relationship between mass pull, concentrate grade, and recovery.

A second gravity concentration test (G-3) was conducted with remaining but recombined 1st and 2nd cleaner tailings by passing the feed once through a Falcon centrifugal gravity concentrator, in a manner to produce a gravity concentrate with a mass pull of approximately 5%.

Results from the initial and bulk gravity concentration tests are shown graphically in Figure 13-9 and in Figure 13-10.

[1] Scavenger gravity programs excluded primary gravity extraction step before flotation

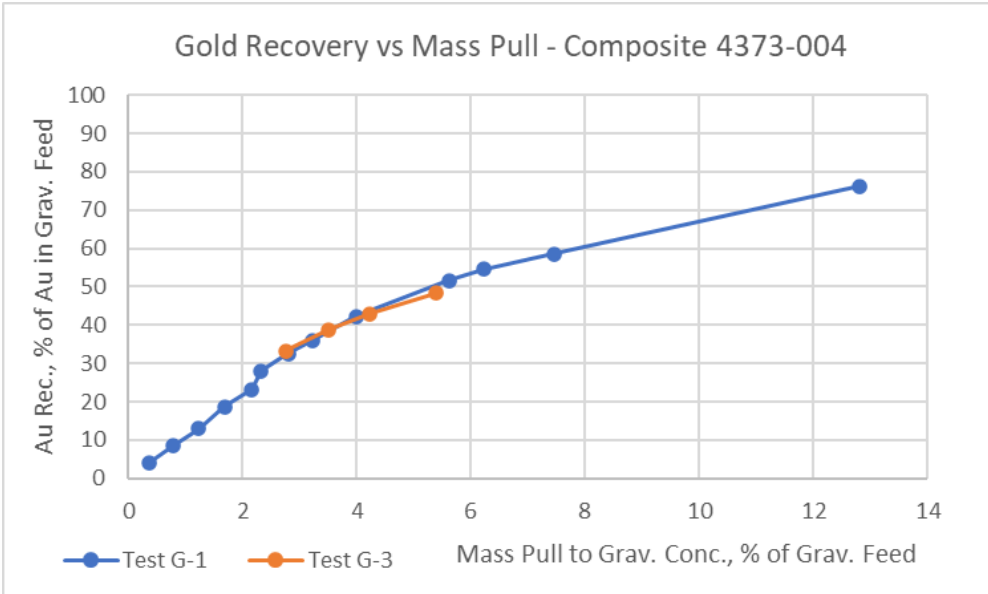


Figure 13-9 Gold Recovery vs Mass Pull, Gravity Concentration, Combined Flotation, Cleaner Tailings, Composite 4373-004, 80%-180µm Feed Size

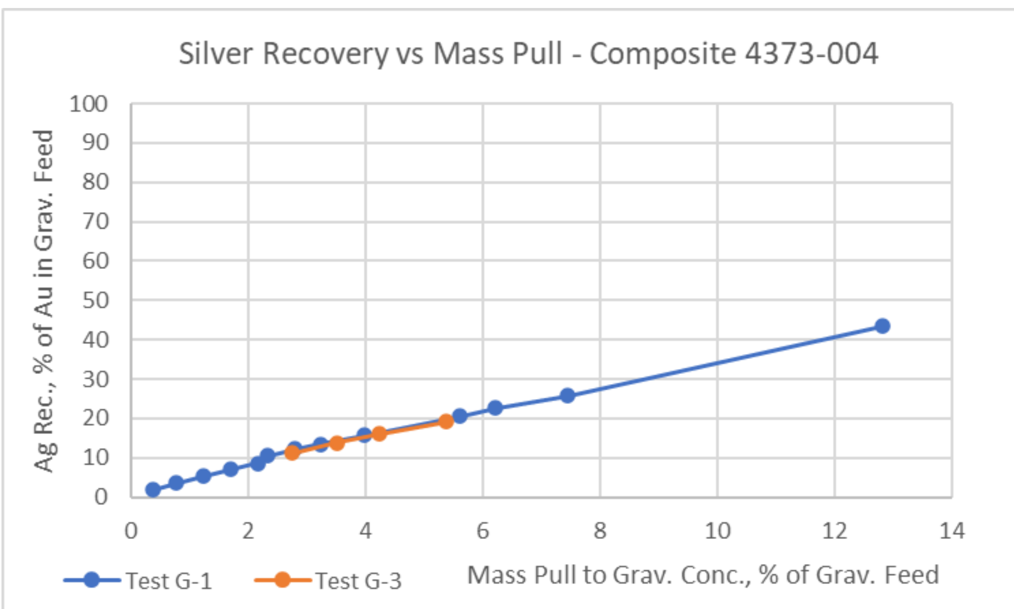


Figure 13-10 Silver Recovery vs. Mass Pull, Gravity Concentration, Combined Flotation, Cleaner Tailings, Composite 4373-004, 80%-180um Feed Size

13.4.6.2.1 Scavenger Gravity Concentration on Rougher Tailings

A scavenger gravity concentration test was conducted on select bulk flotation rougher tailings samples to determine if it is possible to recover particulate gold lost in the rougher tailings. Table

13-17 presents the Scavenger Gravity Concentration Test Results for the Composite 4373-004 Rougher Tailings.

Table 13-17 Scavenger Gravity Concentration Test Results, Composite 4373-004 Rougher Tailings

Test No.	Sample ID	Mass Pull (%)	Head Assay Au (g/t)	Scv. Grav. Conc. Au (g/t)	Scv. Grav. Conc. Au Rec. (%)	Scv. Grav. Tail Au (g/t)	Head Assay Ag (g/t)	Scv. Grav. Conc. Ag (g/t)	Scv. Grav. Conc. Ag Rec. (%)	Scv. Grav. Tail Ag (g/t)
4373-004	G-4	0.07	0.15	102	48.4	0.07	0.2	36.0	11.1	0.2

Note: scv.Gravity conc. Cleaned by panning

Scavenger gravity concentration test results indicated it was possible to recover 48.4% of the residual gold contained in the flotation rougher tailings to a gravity cleaner concentrate. That concentrate was 0.07% of the flotation tailings weight.

Rougher tailings scavenger gravity concentration test results indicate that small amounts of particulate gold contained in the ore reported to the bulk flotation rougher tailings and it could be feasible to recover gold by gravity concentration upstream of the process.

13.4.7 Leaching

This section summarizes the results of the various test programs.

13.4.7.1 G&T Testwork (Project KM2637)

Gravity concentrate leach results from relevant tests for different lithologies are presented in Table 13-17. Cyanide consumptions ranged from 2-7 kg/t. Test 45 and 48 included a 72-hr pre-aeration step and Test 45 saw higher cyanide consumption of 16 kg/t.

Table 13-18 CIL Au & Ag Leach Recovery @24h on Gravity Concentrate – Argillite, Mixed (Argillite/Siltstone), and Siltstone, carbon added (30 g/l).

Test No.	Sample ID	Grind size K80 (um)	Pulp density (%)	DO (mg/l)	Au Rec. @24h (%)	Residue Au (g/t)	Ag Rec. @24h (%)	Residue Ag (g/t)
Argillite								
KM2637-78	872A	37	15	7.4	97.4	0.38	95.5	3.3
KM2637-97	872A	18	9	8.0	97.8	0.39	98.2	1.0
KM2637-82	894	30	15	6.8	97.6	0.67	80.2	2.2
KM2637-99	894	18	9	7.4	98.6	0.25	94.2	0.4
Mixed (Argillite/Siltstone)								
KM2637-83	871	35	16	7.1	97.1	1.07	79.2	2.5
KM2637-100	871	18	9	7.4	97.0	0.61	96.0	0.6

KM2637-81	872B	34	16	7.1	97.6	0.44	74.4	2.5
KM2637-63	891	19	9	7.7	98.0	0.51	--	--
KM2637-79	891	47	17	6.1	97.3	0.55	76.1	2.7
KM2637-98	891	17	9	7.2	98.1	0.45	96.5	1.0
KM2637-45	MC4[a]	20	9	8.3	97.9	0.36	91.4	2.2
KM2637-48	MC4[a]	20	8	5.4	97.4	0.49	--	--
KM2637-60	MC4	20	8	9.4	97.7	0.50	91.4	1.8
Siltstone								
KM2637-80	865-3	20	9	7.1	99.5	0.2	80.6	3.9

a: carbon added Test 45 (22g/l) and Test 48 (10g/l)
MC4: mixed of 871 (23%w/w), 872A (15%), 872B (21%), 891 (23%), 894 (18%)

Average leach recovery of gold from the Argillite was 97.8%, for Argillite/Siltstone was 97.6% and for Siltstone was 99.5%. Recovery of silver for Argillite was 92.0%, for mixed Argillite/Siltstone was 86.4% and for Siltstone was 80.6%. Gravity concentrate mass pull ranged between 1-2.5%.

Flotation concentrate leach results from relevant tests for different lithologies are presented in Table 13-18 through Table 13-20. Cyanide consumptions ranged from 5.5-6.1 kg/t. Low DO levels were achieved using air during the pre-aeration step as the source of oxygen. Cyanide concentration was maintained at 2000ppm during the tests.

Table 13-19 CIL Au & Ag Leach Recovery @48h on Flotation Concentrate (2nd Cl. Conc.), Pre-aerate (72h) – Argillite, and Siltstone/Argillite, carbon added (22 g/ l)

Test No.	Sample ID	Grind size K80 (um)	Pulp density (%)	DO (mg/l)	Au Rec. @48h (%)	Residue Au (g/t)	Ag Rec. @48h (%)	Residue Ag (g/t)	TOC (%)
Argillite									
KM2637-93	872A	18	50	2.0	92.8	0.61	94.7	3.0	0.3
KM2637-95	894	16	50	3.5	97.6	0.31	88.8	1.0	0.2
Siltstone/Argillite									
KM2637-96	871	17	50	3.4	95.7	0.49	90.9	1.0	0.3
KM2637-94	891	19	48	2.3	94.0	1.00	89.1	1.5	0.3

Pre-aerate (72h)

Table 13-20 CIL Au & Ag Leach Recovery @48h and @24h on Flotation Concentrate (1st Cl. Conc.), Pre-aerate (72h) – Argillite, Siltstone and Siltstone/Argillite, carbon added (22 g/l).

Test No.	Sample ID	Grind size K80 (um)	Pulp density (%)	DO (mg/l)	Au Rec. @48h (%)	Residue Au (g/t)	Ag Rec. @48h (%)	Residue Ag (g/t)	TOC (%)
Argillite									
KM2637-72	872A	17	46	3.5	86.4	0.56	88.6	3.5	2.0
KM2637-76	894	16	46	2.5	92.0	0.66	71.0	2.2	1.0
Siltstone									
KM2637-34[b]	865-1	11	33	6.3	90.3[a]	0.32	71.1	2.0	1.0
KM2637-27	865-3	10	25	4.7	92.3[a]	0.46	70.9	2.0	--
KM2637-74	865-3	11	43	1.8	97.2	0.35	79.6	1.9	0.1
Siltstone/Argillite									
KM2637-77	871	16	44	2.5	80.4	2.13	70.7	2.4	1.9
KM2637-75	872B	13	45	2.0	96.7	0.34	75.7	2.1	0.2
KM2637-57	891	12	17	8.0	90.8	0.99	89.6	2.1	1.0
KM2637-62	891	22	50	4.4	84.4	2.37	76.1	2.1	1.0
KM2637-73	891	12	43	1.4	88.6	1.32	67.7	2.1	1.1
KM2637-44	MC4	12	17	8.2	85.9[a]	0.91	78.6	1.9	0.8
KM2637-47	MC4	14	19	5.7	80.1[a]	1.47	--	--	0.8
KM2637-59	MC4	23	50	6.2	85.5	1.31	79.1	2.1	1.0
KM2637-65	MC4	11	33	5.7	93.3	0.65	75.9	2.5	1.1

a: leach time 24h
b: oxygen purge

Average gold recovery of leaching of flotation concentrates (1st Cl. Conc.) @48h was 89.2% for Argillite, 97.2% for Siltstone and 88.5% for Siltstone/Argillite. Average silver recovery from Argillite samples was 79.8%, for Siltstone was 79.6% and for Siltstone/Argillite was 76.4%. TOC content was high in the 1st Cleaner concentrate samples (Table 13-20), with improved gold leach recoveries ranging between 93-97% seen at TOC concentrations <0.5% (Table 13-19).

13.4.7.2 SGS Testwork (Project 12488-002)

Results from selected relevant tests for the different lithologies are presented in Table 13-21 through Table 13-23.

Table 13-21 CIL Au & Ag Leach Recovery @24h on Gravity Concentrate – Argillite, Siltstone and Tuff, carbon added (22 g/l), 2g/l NaCN added

Test No.	Sample ID	Grind size K80 (um)	Pulp density (%)	Au Rec. @24h (%)	Residue Au (g/t)	Ag Rec. @24h (%)	Residue Ag (g/t)	DO avg (mg/l)
Argillite								
CIL 33	SM1	16	27	97.5	0.40	90.9	0.8	8.1
CIL 2	SM2	14	34	90.3	2.80	94.5	<0.5	--
CIL 4	SM6	15	35	93.2	3.57	98.0	0.8	--
CIL 7	SM9	14	38	90.2	1.89	91.5	0.7	--
CIL 8	SM10	12	36	97.6	0.29	91.5	<0.5	--
CIL 35	SM14	17	36	89.7	1.66	96.5	<0.5	8.1
CIL 12	SM18	15	43	88.2	1.33	86.1	0.6	--
CIL 13	SM19	16	43	96.2	3.28	76.9	6.4	--
CIL 14	SM20	15	40	90.0	3.64	94.7	<0.5	--
CIL 15	SM21	13	42	92.6	0.82	13.1	1.2	--
Siltstone								
CIL 10	SM12	12	39	91.2	0.98	89.3	<0.5	--
CIL 16	SM24	14	40	79.9	3.19	86.9	0.8	--
CIL 36	SM24	13	29	76.9	3.03	84.0	1.0	--
Tuff								
CIL 3	SM3	12	42	97.8	0.31	22.0	0.7	--
CIL 5	SM7	11	41	91.4	1.13	92.2	<0.5	--
CIL 6	SM8	14	41	98.0	0.30	88.4	0.8	--
CIL 9	SM11	--	44	98.0	0.12	69.6	3.8	--

Average gold recovery of the leaching of gravity concentrates was 92.6% for Argillite, 82.7% for Siltstone and 96.3% for Tuff. Average silver recovery from Argillite samples was 91.2%, for Siltstone was 86.7% and for Tuff was 83.4%. These tests were performed without a pre-aeration step and DO was not added nor were DO levels measured.

Table 13-22 Intensive Cyanidation Au & Ag Leach Recovery @24h on Gravity Concentrate, Argillite, NaCN (10 g/l), carbon added (22 g/l)

Test No.	Sample ID	Grind size K80 (um)	Pulp density (%)	Au Rec. @24h (%)	Residue Au (g/t)	Ag Rec. @24h (%)	Residue Ag (g/t)	DO (mg/l)
CIL 37	SM2	13	36	84.4	3.21	93.2	0.8	30.7
CIL 38	SM9	13	33	83.2	3.86	86.0	1.6	8.3
CIL 39	SM20	19	33	87.1	2.21	87.8	0.8	37.2

Gold recovery from intensive cyanidation testwork (Table 13-22) showed lower recoveries than the same samples using 2.0 g/l of NaCN (Table 13-21). For SM2 sample gold recovery was 84.4% compared to 90.3%, for SM9 sample gold recovery was 83.2% compared to 90.2% and for SM20 sample gold recovery was 87.1% compared to 90.0%. Repeat tests on the leach residues using the same cyanide concentration and with oxygen addition during leaching showed improved results, with up to a 6% increase in recovery.

Table 13-23 CIL Au & Ag Leach Recovery @48h on Flotation Concentrate (3rd Cl. Conc.) – Argillite, Siltstone and Tuff, carbon added (22 g/l)

Test No.	Sample ID	Grind size K80 (um)	Pulp density (%)	Au Rec. @48h (%)	Residue Au (g/t)	Ag Rec. @48h (%)	Residue Ag (g/t)	Pre-aeration (h)	Pre-aeration DO (mg/l)	Cyanidation DO (mg/l)	TOC (%)	Calc leach Feed Au (g/t)
Argillite												
CIL 17	SM1	15	39	95.8	0.53	88.2	1.4	21	36.6	7.3	0.2	12.5
CIL 18	SM2	12	38	97.5	0.36	85.3	1.8	24	21.0	4.2	0.4	14.6
CIL 20	SM6	46	36	97.1	0.69	87.8	1.5	21	40.2	7.2	0.2	23.5
CIL 23	SM9	109	40	91.7	0.89	83.3	3.2	5	24.6	8	0.6	10.6
CIL 24	SM10	17	35	96.8	0.29	84.3	1.9	17	12.1	7.2	0.4	8.8
CIL 27	SM14	10	42	96.6	0.35	88.3	0.7	5	0.3	6.5	0.5	10.1
CIL 28	SM18	10	17	94.0	0.46	74.5	2.3	22	30	8.3	0.4	7.6
CIL 29	SM19	11	36	98.6	0.29	83.9	2.4	20	0.6	3.6	0.5	19.8
CIL 30	SM20	9	33	97.8	0.32	77.2	2.2	20	31.5	4.5	0.6	14.5
CIL 31	SM21	16	29	97.9	0.24	88.5	2.7	5	23	7.4	0.9	11.6
Siltstone												
CIL 26	SM12	11	37	94.8	0.52	82.1	1.9	20	5.1	5.4	0.5	10.1
CIL 32	SM24	16	32	84.9	2.07	76.0	3.8	21	21.3	7.6	0.7	13.7
Tuff												
CIL 19	SM3	20	33	87.1	1.05	80.4	3.0	21	21.4	7.5	0.4	8.2
CIL 21	SM7	34	39	95.7	0.43	86.1	1.9	5	30.1	8.2	0.4	9.9
CIL 22	SM8	14	32	91.1	1.04	81.6	4.0	5	30.1	7.5	0.6	11.6
CIL 25	SM11	15	33	95.1	0.48	81.4	5.6	20	41.5	7.4	0.3	9.7

Average gold recovery of leaching flotation concentrates was 96.9% for Argillite, 89.8% for Siltstone and 92.3% for Tuff. Average silver recovery from Argillite samples was 84.2%, for Siltstone was 79.1% and for Tuff was 82.4%. TOC concentration was less than 1% for all tests.

13.4.7.3 Met-Solve Testwork (Project MS1735)

Table 13-24 presents CIL gold and silver recovery in the Flotation Concentrate.

Table 13-24 CIL Au & Ag Recovery @48h on Flotation Concentrate (2nd Cl. Conc.) – Siltstone, NaCN (2 g/l), carbon added (20 g/l), 12h Pre-aeration

Test No.	Sample ID	Grind size K80 (um)	Pulp density (%)	Au Rec. @48h (%)	Residue Au (g/t)	Ag Rec. @48h (%)	Residue Ag (g/t)	Leach Feed, Calc. Au (g/t)
ND221	865-3	20	52	98.4	0.46	97.2	0.5	30.2

13.4.7.4 McClelland Testwork (Project MLI4373)

Table 13-25 presents CIL leach recovery for gold and silver on the Flotation Concentrate.

Table 13-25 CIL Au & Ag Leach Recovery @48h on Flotation Concentrate (2nd Cl. Conc.) – Siltstone/Argillite, carbon added (20 g/l), 12h Pre-aeration, 5ppm DO

Test No.	Sample ID	Grind size K80 (um)	Pulp density (%)	Au Rec. @48h (%)	Residue Au (g/t)	Ag Rec. @48h (%)	Residue Ag (g/t)	Leach Feed, Calc. Au (g/t)
Siltstone/Argillite								
4373-004	CY-01	80%-75um	50	90.8	2.59	78.6	<3	28.1
4373-004	CY-02	96%-37um	50	99.8	0.07	78.6	<3	29.1
4373-004	CY-03	97%-25um	50	98.8	0.32	83.3	<3	27.6

Scoping CIL cyanidation testing was conducted on flotation recleaner concentrate produced during bulk testing on composite 4373-004, to confirm amenability to cyanidation processing and to optimize regrind size for the concentrate. Tests were conducted without regrinding and at 96%-37um and 97%-25um regrind sizes. CIL recovery, after 48 hours of leaching, improved from 90.8% to 99.8% of gold contained in the flotation concentrate by regrinding to 96%-37um. Reagent consumption was 1.1-3.3g/t for NaCN and 1.1-5.6kg/t for Lime.

Comparing Table 13-19 and Table 13-25 leaching of flotation concentrates (2nd Cl. Conc.) @48h with similar carbon added (20-22 g/l) and same lithology (Siltstone/Argillite), the gold recovery was higher (98.8%) in the McClelland testwork compared to 95% average leach gold recovery in the G&T testwork, possibly attributable to higher DO levels in the McClelland testwork.

Table 13-26 CIL Au & Ag Leach Recovery @48h on Flotation Concentrate (2nd Cl. Conc.) + Scv. Grav. Conc., – 4373-Al-1, Siltstone/Argillite, carbon added (20 g/l).

Test No.	Sample ID	Grind size K80 (um)	Pulp density (%)	Au Rec. @48h (%)	Residue Au (g/t)	Ag Rec. @48h (%)	Residue Ag (g/t)	Leach Feed, Calc. Au (g/t)
4373-AL-1	FT-03	95%-37um	50	97.3	0.63	80.2	3.0	23.7

Recovery results of gold leaching of a blend of flotation concentrates and scavenger gravity concentrates Table 13-26 was lower (97.3%) than only leaching the flotation concentrates (99.8%) (Table 13-24).

CIL cyanidation testing was conducted on final (flotation recleaner + gravity) concentrate produced from the bulk process simulation test conducted on composite 4373-004, to confirm amenability to agitated cyanidation. A CIL recovery of 97.3% of the gold contained in the combined concentrate was obtained, using a 95%-37 um regrind size, with a 48-hour leach cycle. Cyanide and lime consumption from the bulk leach test was 3.3 kg/t NaCN and 1.1 kg/t for lime.

13.4.8 Cyanide destruction

Cyanide destruction was investigated by McClelland.

Cyanide destruction test work was carried out on leach residues from the combined bulk concentrate (flotation/gravity) composite 4373-004 tests using the SO_2/air process. Test conditions included a sample mass of 6.7 kg, grind size (95%-37 um), pulp density 50% solids, NaCN concentration of 2 g/l, pH of 9.4-11 with lime addition, carbon addition of 20 g/l, and a leaching time of 48 hours.

Test results are shown in Table 13-27. Cyanide detoxification testing showed that an SO_2 addition rate of 6.0 g/g CNWAD and Ca(OH)2 addition rate of 4.2 g/g CNWAD was required to achieve a treated slurry CNWAD concentration of 1.50 mg/l. Copper addition was not required.

Table 13-27 SO2/Air Detoxification Test Results, CIL/Cyanidation Slurry (Test AL-1), Combined Bulk Concentrate (Flotation/Gravity), Composite 4373-004, 95%-37µm (Regrind (CIL Feed) Size.

Composite	Detoxification Test	Residence Time, hours	SO_2 Addition, gSO_2/gCN_{wad}	Cu^{2+} Addition, mg/L	Lime Consumption, $gCa(OH)_2$/g CNwad	Slurry pH	Solution Analysis		
							CN_{wad} mg/L	Cu mg/L	Fe mg/L
4373-FT03	SO2-6 (Feed)	---	---	---	---	10.8	497.10	670	34.0
4373-FT03	SO2-6 (Treated)	2.0	6.0	0	4.22	8.5	1.50	1.91	0.12

13.5 Recovery Estimate

A recovery model for gold and silver was developed from the recent McClelland (Project 4373-002/003), SGS (Project 12488-002), and G&T (Project KM2637) laboratory test work programs to estimate recoveries from feed materials representing the average feed grades from the open pit of the deposit. The recovery estimate is built up using stage recoveries for the selected unit operations.

13.5.1 Primary Gravity Circuit Recovery

The EGRG test data represents the theoretical maximum gravity circuit recovery. Bench-scale test results are typically derated when predicting commercial scale plant performance. The basis for selection of the primary gravity circuit recovery of 18% when treating 32% of the mill circulating load is the modelling by Knelson presented in Section 13.4.5.1.

13.5.2 Flotation Circuit Recovery

Flotation data for the rougher and cleaner circuits was analysed and normalised to a concentrate grade of 10 g/t Au. The recovery versus head grade model parameters are calculated by minimizing the squared sum of errors (SSE) between the test work points and the modelled results using a regression function (Figure 13-11).

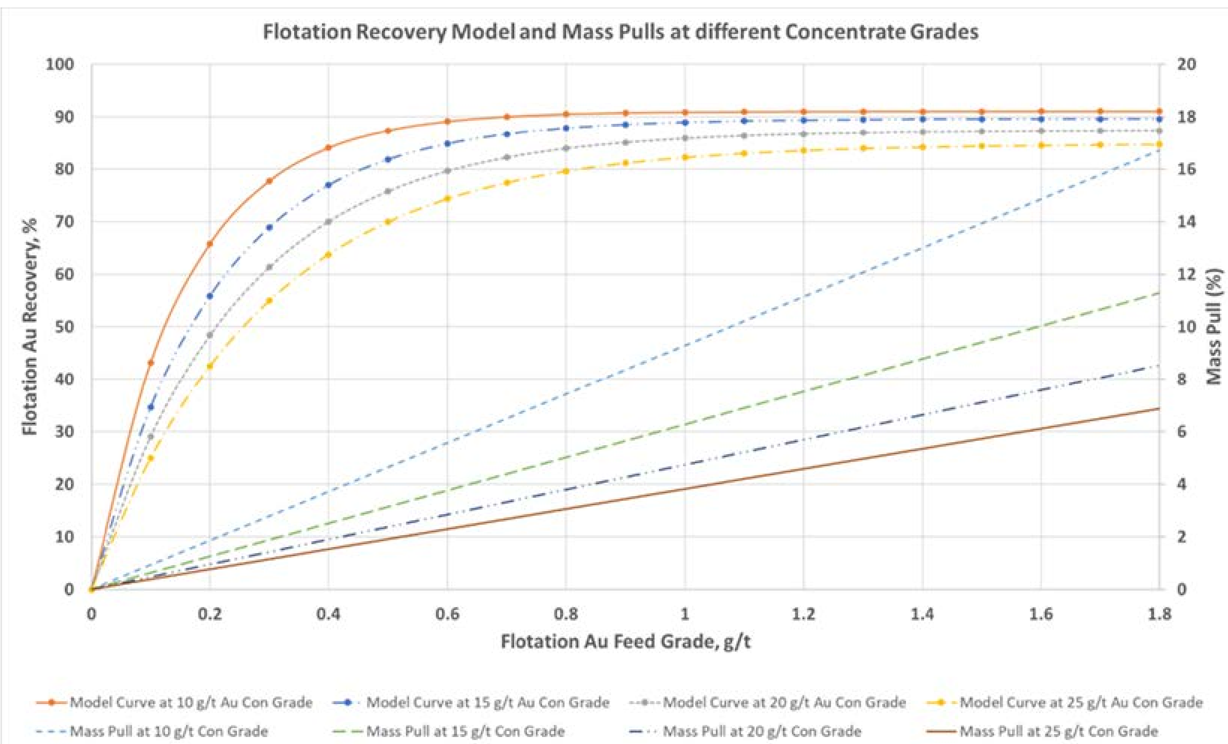


Figure 13-11 Flotation Recovery and Mass Pull at different concentrate grades

A concentrate grade of 10 g/t Au was selected for recovery estimation (Figure 13-12).

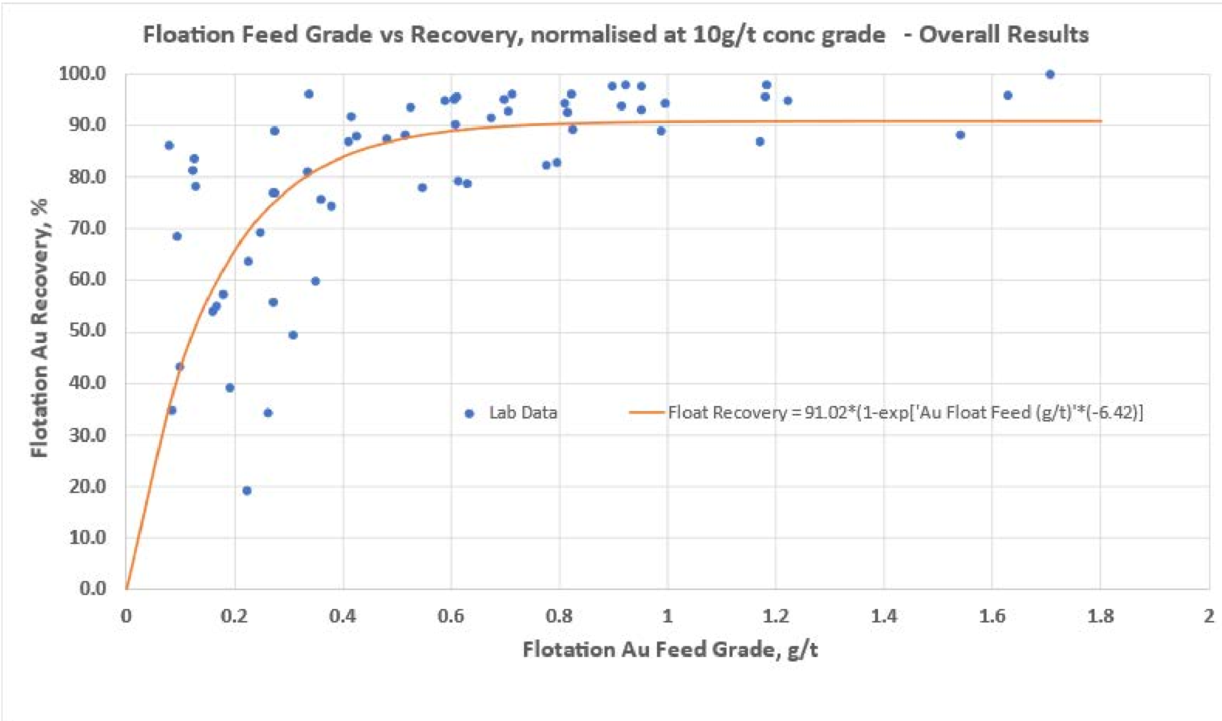


Figure 13-12 Gold Feed Grade and Gold Recovery Relationship at 10 g/t Au grade concentrate

Flotation circuit gold recovery is estimated using the following relationship:

$$\boldsymbol{Flotation\ Recovery} = \mathbf{91.02} * (\mathbf{1} - \boldsymbol{EXP}(-\mathbf{6.42} * \boldsymbol{Flotation\ Feed\ grade}))$$

Flotation circuit silver recovery is estimated based on the relationship of gold to silver recovery shown in Figure 13-13.

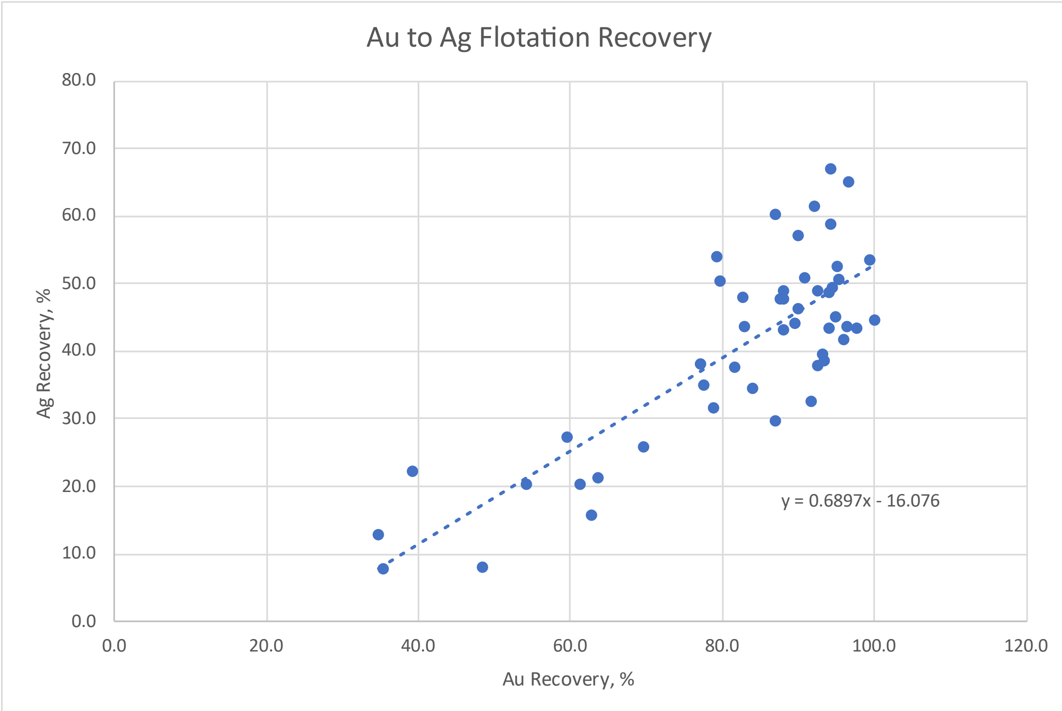


Figure 13-13 Relationship of Gold and Silver Flotation Recovery

Flotation circuit mass recovery versus Gold feed grade is shown in Figure 13-14 and mass recovery versus Sulphur feed grade is shown in Figure 13-15.

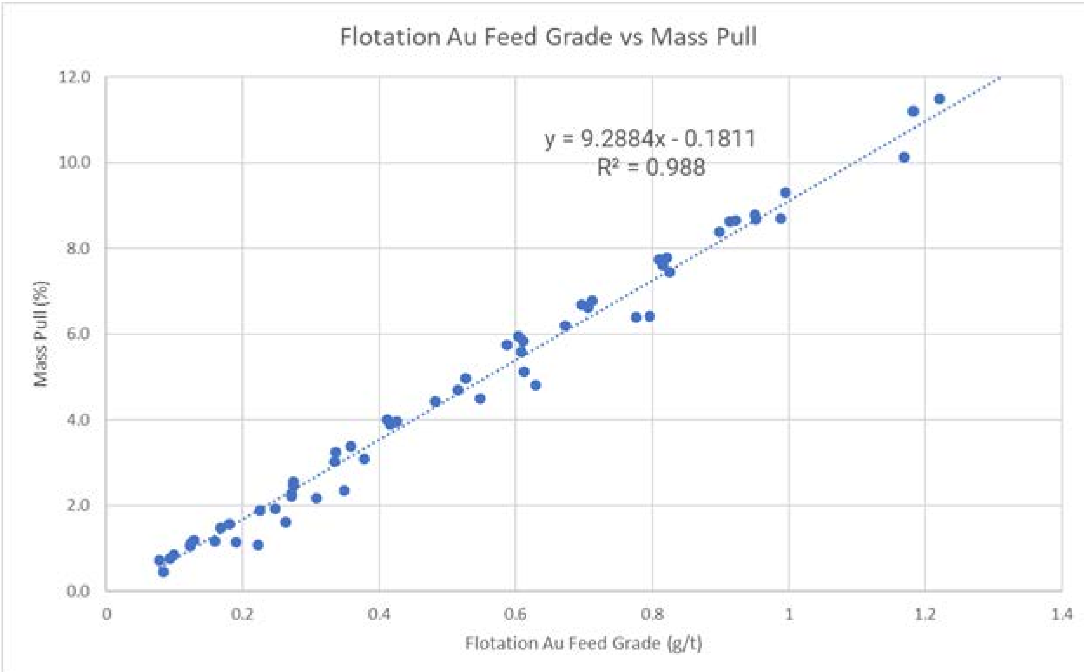


Figure 13-14 Gold Recovery and Flotation Mass Pull Relationship at 10 g/t Au grade concentrate

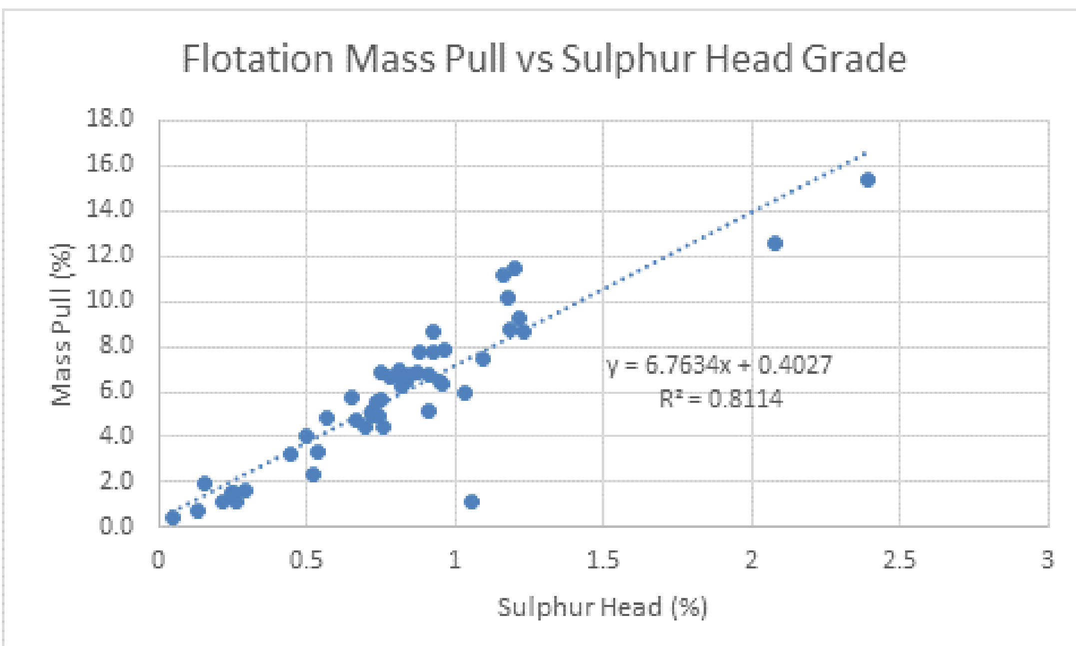


Figure 13-15 Sulphur Head Grade and Flotation Mass Pull Relationship at 10 g/t Au grade concentrate

At the average LOM gold head grade of 0.77g/t, flotation circuit mass recovery of approximately 6% is expected from Figure 13-14. The average LOM S head grade of 1.3% corresponds to approximately a 3% mass pull to final concentrate, as shown in Figure 13-15. The plant design criteria are based on a flotation circuit mass pull of 3%. While this is lower than the mass predicted when correlating to gold recovery, the lower mass pull is considered achievable using DFR technology. Pilot and operational benchmark data provided by Woodgrove Technologies shows improved cleaner circuit performance (higher concentration upgrade ratios) when using DFR flotation cells due to their increased froth recovery and froth washing function. Pilot scale testwork is planned of the next stage of project development to demonstrate the DFR cell performance in this application (ability to reject organic carbon and achieve high recovery gold to a low mass of concentrate). There is a risk that the downstream concentrate CIL circuit is undersized in this study and this risk will be addressed through ongoing testwork.

13.5.3 Scavenger Gravity Circuit Recovery

Combined Flotation cleaner circuit tailings (cleaner and recleaner tailings) gold content is estimated as a function of flotation feed grade from tests SM2,6,19,20 and ND200 as most representative for selected plant operating conditions. The relationship is shown below in Figure 13-16.

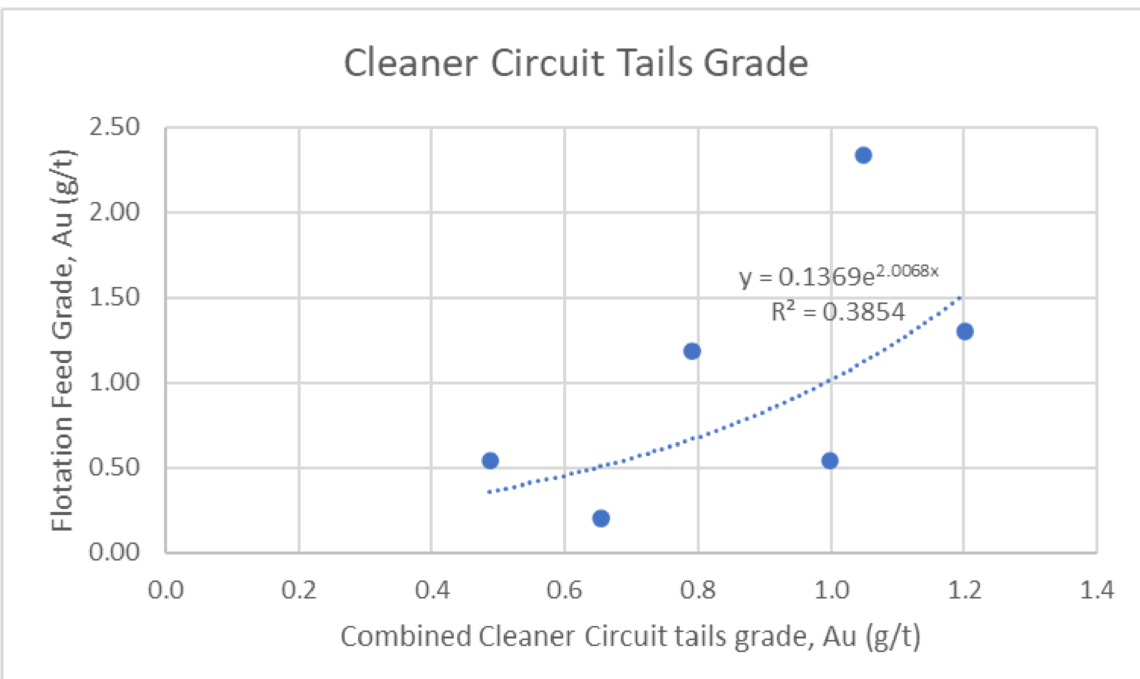


Figure 13-16 Flotation cleaner circuit tailings grade as a function of flotation feed grade

Laboratory scale scavenger gravity circuit gold recovery of 50% at a mass pull of 5% (from Figure 13-9) is de-rated by 50% to allow for gravity gold recovery in the primary circuit as well as on the advice of a gravity concentrator vendor.

The gravity scavenger circuit contributes 1% in overall recovery for the range of head grades in the mine plan.

13.5.4 Leach Circuit Recovery

A gold recovery of 98.5% has been applied to intensive leaching of gravity concentrates, in line with industry benchmarks for low gravity circuit mass pull (<0.05%) and using high cyanide concentrations and peroxide or LeachAid.

Leach gold recovery has been estimated from the SGS and Met-Solve and McClelland test results employing regrind, from Table 13-23, Table 13-24 and Table 13-25 and is shown as a function of leach feed gold grade in Figure 13-17.

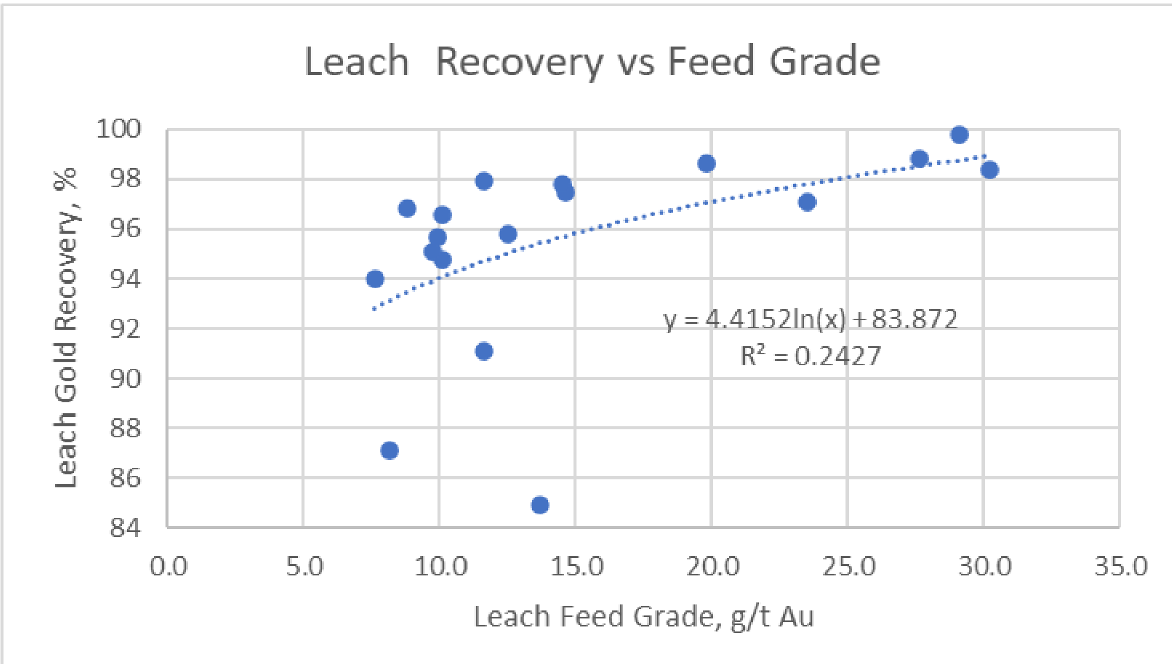


Figure 13-17 Leach gold recovery as a function of leach feed grade

A leach recovery of 80% for silver has been applied for flotation concentrate CIL following regrind to a P_{80} of 22 µm.

13.5.5 Overall Plant Recovery

Combination of the recovery factors per unit operation results in the following relationships which have been used to estimate plant recovery, applicable over the range of head grades evaluated (Figure 13-18 and Figure 13-19):

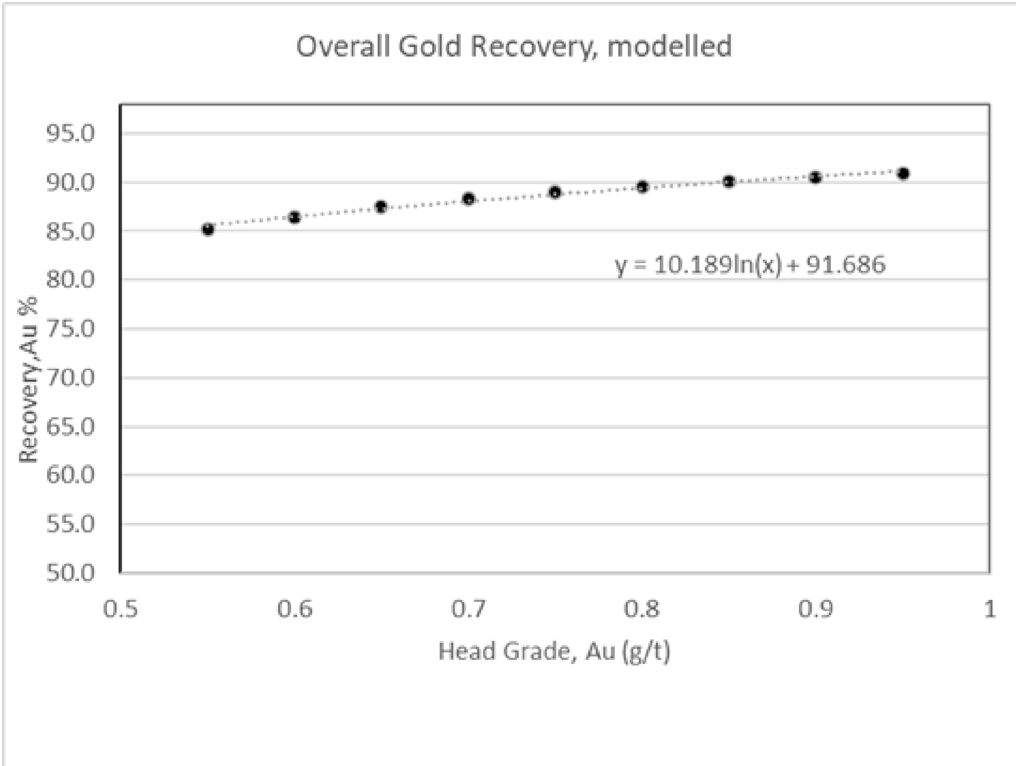


Figure 13-18 Overall Gold Recovery as a Function of Head Grade

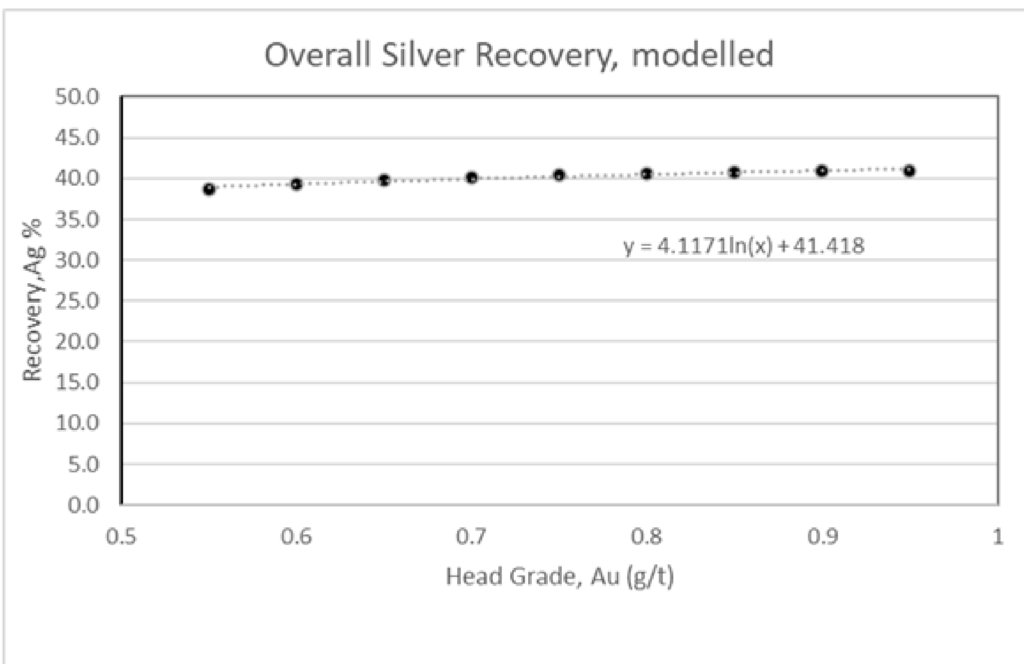


Figure 13-19 Overall Silver Recovery as a Function of Head Grade

13.6 Discussions

The metallurgical testwork data used to develop the flowsheet and process design parameters included a review of historical test work and the latest mine plan which was prepared on December 14, 2020.

Analysis of the data, which included primary gravity concentration, flotation, cyanidation, and gravity concentration on the cleaner tailings, showed that primary gravity concentration was not considered as part of the process flowsheet.

The original analyses showed there was no benefit in keeping primary gravity concentration, at the gold price prevailing at the time, however after further analysis and considering the McClelland test results where scavenging gravity concentration of the cleaner & recleaner tailings and rougher flotation tailings, showed there was gravity recoverable gold downstream. It was determined, if there was gravity recoverable gold downstream then it would be feasible to recover the gold in an upstream process.

There were three main changes to the process flowsheet from the 2019 PEA.

- Primary, secondary, and tertiary crushers were replaced by a SABC grinding circuit
- Primary gravity concentration was included as part of the grinding circuit together with the intensive leach reactor
- Conventional flotation cells were replaced by DFR flotation technology.

13.7 Conclusions

Spanish Mountain Gold completed numerous comminution and metallurgical testwork programs to support the PFS. This included head grade analyses, a full suite of comminution tests, flotation, gravity separation, and leach tests and cyanide detoxification.

Tests were performed on mineralization that is considered representative of the 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 over the life of mine. The grade variability samples gold and silver grades ranged from 0.15–3.25 g/t Au and 0.5–4.4 g/t Ag.

Comment re primary grind:

Selection of primary grind size should be revisited in future studies as the P80 size of 180 µm was selected at a time when the lower gold price did not justify finer grinding.

Bulk mineralogy on select composites showed that pyrite was the main sulfide mineral present, representing between 0.5-2.5% by mass. Sphalerite and chalcopyrite were also present in relative order of abundance.

Comminution testing showed that the materials tested are highly variable in competency. The breakage data showed the ore can be classified as competent and moderately hard, and moderately abrasive with SAG Mill Comminution (SMC) tests ranging from 26-51.7. Conventional Bond tests showed significant variation in hardness, with Bond rod mill work indices ranging 12.4-17.5 kWh/t and Bond ball mill work indices ranging from 10.9-16.7 kWh/t.

Rougher flotation tests showed high sulfide recovery was generally achieved within 8 minutes of flotation time. Flotation recoveries to cleaner concentrate ranged 80-92% for gold, 25-55% for silver.

High leach recoveries were achieved when leach feed was reduced to <0.5% TOC and after regrind.

Overall plant recoveries for gold are predicted to range from 85–92% for head grades ranging from 0.6-1 g/t Au. Overall plant recoveries for silver are predicted to range from 38–42%.

Cyanide detoxification tests reduced WAD cyanide levels to 1.5 mg/L with moderate reagent consumption rates.

The primary gravity concentration circuit was reinstated as part of the process flowsheet optimization.

14 MINERAL RESOURCE ESTIMATE

This Technical Report presents an update to the Mineral Resource estimate for the Spanish Mountain Gold Project as part of the Prefeasibility Study.

The Mineral Resource estimate for the Spanish Mountain Gold Project was prepared by Independent Qualified Person, Marc Jutras, P.Eng., M.A.Sc., Principal, Ginto Consulting Inc. The Mineral Resource estimate has been classified as “Measured”, "Indicated" and "Inferred" according to the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) "CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines" (May 2014). The Mineral Resource estimate has an effective date of February 3, 2021. Marc Jutras visited the property on November 25, 2020, at which time he inspected the core logging, core cutting, and core storage facilities. Logging and sample preparation procedures were examined, as well as the drilling of a reverse circulation hole. The geology office was visited, and the acquisition and storage of geologic data were discussed.

For the 2017 PEA a three-dimensional geologic model was produced by SMG using Vulcan 3D mining software. The main zone mineralization was modelled into an Upper Argillite unit, an Altered Siltstone unit, a Tuff unit, and a Lower Argillite unit. The North Zone Argillite was a separate solid.

The Mineral Resource estimate of the Spanish Mountain Gold Project’s Prefeasibility Study (PFS) represents an update of the 2019 Preliminary Economic Study (PEA). Although there are no new drillholes added, the grade estimation strategy has been revised for the PFS.

The geology model previously used for the 2019 PEA study, was retained as the geology model for the PFS as no new drillhole data was available. The exploratory data and variographic analyses, grade estimation, and Mineral Resource classification, were revisited for the PFS. In addition to grade estimates for Au and Ag, grade estimates for As, Ca, and total S were provided in this study.

This Mineral Resource estimation exercise was primarily undertaken with the Vulcan® software and utilities internally developed in GSLIB-type format. The following sections outline the procedures undertaken to calculate the Mineral Resource.

14.1 Drillhole Data

The drillhole database was provided by Spanish Mountain Gold with a cut-off date of September 29, 2018. It is comprised of 1,003 surface holes with a total of 127,464 assays for gold (Au) in g/t, 125,804 assays for silver (Ag) in g/t, 124,695 assays for arsenic (As) in ppm, 125,552 assays for calcium (Ca) in %, and 56,848 assays for total sulphur (S) in %. The Au and Ag assays were utilized in the estimation of the Mineral Resources, while the As, Ca, and S assays were utilized to produce estimates for metallurgical and environmental purposes. All drillhole data is derived from drilling between 2004 and 2018. Although a drilling campaign was completed in late 2020, assays were not available for the current study. A verification of the drillhole database was carried out by Marc Jutras as an independent validation of the drillhole data (see Section 12 – Data Verification).

14.1.1 Drillhole Data Statistics

General statistics of the collar, survey, and assay files are presented in Table 14-1. Statistics on the drillhole database performed by year, type and size are presented in Table 14-2 and Table 14-3, respectively.

As seen in Table 14-1, the average drillhole depth is 209.5m, with depths varying from 7.6m to 646.2m. Sample lengths are observed to be 1.58m on average, with samples lengths varying from 0.12m to 12.48m, and with the most common sampling length being 1.50m for approximately 75% of the samples.

Gold grade statistics of the original samples are presented in Table 14-4 at various cut-off grades. It can be seen from this Table that the meters display a sharp decrease with elevated cut-off grades, while the accumulation (grade x thickness) of gold has a more consistently decreasing pattern with elevated cut-off grades. This observation seems to indicate higher grades from fewer samples. This can also be noticed in the fact that the average grades are much higher than the cut-off grades.

Table 14-1 Statistics on the Drillhole Database

Collar Data	Number of Data	Mean	Standard Deviation	Coefficient of Variation	Minimum	Lower Quartile	Median	Upper Quartile	Maximum	Number of 0.0 Values	Number of < 0.0 Values
Easting (X)	1003	603997.0	861.987	0.001	600227.0	603991.0	604301.0	604460.0	605373.0	-	-
Northing (Y)	1003	827963.0	737.608	0.001	824600.0	827580.0	827929.0	828377.0	830437.0	-	-
Elevation (Z)	1003	1096.06	111.449	0.102	907.93	997.83	1091.13	1185.69	1430.0	-	-
Hole Depth	1003	209.538	102.432	0.489	7.62	129.69	199.64	293.07	646.18	-	-
Azimuth	1003	92.572	83.559	0.903	0.0	0.0	118.0	130.0	360.0	-	-
Dip	1003	-71.983	13.901	-0.193	-90.0	-90.0	-70.0	-50.0	-45.0	-	-
Overburden	1003	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	-	-
Survey Data										-	-
Azimuth	3149	156.455	84.282	0.539	0.0	110.68	124.5	215.92	359.83	-	-
Dip	3149	-72.628	11.943	-0.164	0.0	0.0	0.0	0.0	0.0	-	-
Assay Data											
Interval Length (from-to)	127474	1.563	0.356	0.228	0.12	1.5	1.5	1.5	12.48	0	0
AU_GPT	127474	0.236	1.604	6.786	0.0	0.02	0.04	0.16	241.0	10	1211
AG_GPT	127474	1.903	76.6	40.254	0.0	0.25	0.3	0.6783	18135.0	1678	1203

Table 14-2 Drillhole Database – Statistics by Year

Spanish Mountain Drillhole Database						% of Total			
Year	# Holes Drilled	Length Drilled (m)	# Assay Intervals	Total Length Assayed (m)	% Assayed	# Holes Drilled	Length Drilled	# Assay Intervals	Total Length Assayed
2004	41	3,437.8	1,847	3,086.6	89.8%	4.1%	1.6%	1.4%	1.5%
2005	64	11,122.1	7,303	10,704.6	96.6%	6.4%	5.3%	5.7%	5.4%
2006	148	28,992.9	18,168	27,457.3	94.7%	14.8%	13.8%	14.3%	13.8%
2007	125	28,323.6	17,059	26,775.0	94.5%	12.5%	13.5%	13.4%	13.4%
2008	160	40,393.4	25,047	39,148.3	96.9%	16.0%	19.2%	19.7%	19.7%
2009	59	13,545.6	8,282	12,939.8	95.5%	5.9%	6.4%	6.5%	6.5%
2010	20	6,835.3	3,636	5,585.8	81.7%	2.0%	3.3%	2.9%	2.8%
2011	129	33,236.6	19,909	31,288.5	94.1%	12.9%	15.8%	15.6%	15.7%
2012	169	30,733.7	17,630	29,171.5	94.9%	16.8%	14.6%	13.8%	14.6%
2013	56	9,229.1	5,936	9,022.1	97.8%	5.6%	4.4%	4.7%	4.5%
2014	18	2,676.1	1,669	2,531.1	94.6%	1.8%	1.3%	1.3%	1.3%
2018	14	1,640.1	978	1,492.2	91.0%	1.4%	0.8%	0.8%	0.7%
Total	1,003	210,166.3	127,464	199,202.8	94.8%	100.0%	100.0%	100.0%	100.0%

Table 14-3 Drillhole Database – Statistics by Type

	Number of Drillholes					Number of Meters Drilled					% of Total			
Year	HQ	NQ	PQ	RC	Total	HQ	NQ	PQ	RC	Total	HQ	NQ	PQ	RC
2004	0	7	0	34	41	0.0	934.2	0.0	2,503.6	3,437.8	0.0%	27.2%	0.0%	72.8%
2005	0	34	0	30	64	0.0	7,745.2	0.0	3,376.9	11,122.1	0.0%	69.6%	0.0%	30.4%
2006	0	102	0	46	148	0.0	24,072.5	0.0	4,920.4	28,992.9	0.0%	83.0%	0.0%	17.0%
2007	0	125	0	0	125	0.0	28,323.6	0.0	0.0	28,323.6	0.0%	100.0%	0.0%	0.0%
2008	0	160	0	0	160	0.0	40,393.4	0.0	0.0	40,393.4	0.0%	100.0%	0.0%	0.0%
2009	32	27	0	0	59	4610.8	8,934.7	0.0	0.0	13,545.6	34.0%	66.0%	0.0%	0.0%
2010	12	8	0	0	20	3361.1	3,474.2	0.0	0.0	6,835.3	49.2%	50.8%	0.0%	0.0%
2011	4	125	0	0	129	960.5	32,276.1	0.0	0.0	33,236.6	2.9%	97.1%	0.0%	0.0%
2012	32	135	2	0	169	5701.0	24,915.6	117.0	0.0	30,733.7	18.5%	81.1%	0.4%	0.0%
2013	0	0	0	56	56	0.0	0.0	0.0	9,229.1	9,229.1	0.0%	0.0%	0.0%	100.0%
2014	0	0	0	18	18	0.0	0.0	0.0	2,676.1	2,676.1	0.0%	0.0%	0.0%	100.0%
2018	3	0	0	11	14	548.9	0.0	0.0	1,091.2	1,640.1	33.5%	0.0%	0.0%	66.5%
Total	83	723	2	195	1,003	15182.5	171,069.5	117.0	23,797.3	210,166.3	7.2%	81.4%	0.1%	11.3%

Table 14-4 Statistics of Gold Grades from Original Assays

Statistics of Gold Assays Above Cut-Off								
Cut-Off g/t	Total Meters	Increment. Percent	Avg. Au g/t	Grade-thickness g/t-m	Increment Percent	Std. Dev.	Coefficient. of Variation	# of Samples
0.00	199,202.8	100.0	0.23	45,816.6	100.0	1.60	6.76	127,464
0.25	35,380.5	17.8	1.02	36,088.1	78.8	3.66	3.42	23,109
0.50	18,932.5	9.5	1.60	30,292.0	66.1	4.90	2.90	12,459
0.75	12,207.9	6.1	2.15	26,247.0	57.3	5.99	2.64	8,100
1.00	8,491.1	4.3	2.72	23,095.8	50.4	7.07	2.49	5,690
2.50	2,058.7	1.0	6.54	13,463.9	29.4	13.2	1.91	1,439
5.00	632.7	0.3	13.71	8,674.3	18.9	21.3	1.49	471

14.1.2 Location, Orientation, and Spacing of Drillholes

The location of the drillholes is presented in Figure 14-1 for the Spanish Mountain Gold Project area. As seen in this Figure, although a large proportion of the drillholes are located within the area of interest, identified as the block model's limits (in blue), drillholes in surrounding areas are also observed. The latter are however not included as part of the current resource estimate.

Statistics on drillhole spacing are presented in Table 14-5 for each mineralized zone within the area of interest. The overall average drillhole spacing is 30.0m while the overall median drillhole spacing is 23.2m. These results indicate an adequate drillhole spacing in the area of interest for the calculation of a Mineral Resource estimation.

Regarding the orientation of the drillholes, five main orientations of drilling are noted: to the northeast at azimuths varying from 15° to 35° and dips from -40° to -85°, to the southeast at azimuths varying from 110° to 130° and dips from -55° to -85°, to the southwest at azimuths varying from 180° to 220° and dips from -45° to -85°, to the northwest at azimuths varying from 295° to 330° and dips from -55° to -85°, and vertical. Figure 14-2, which represents the bottom half of a sphere, displays the various azimuth and dip angles of the drillholes for the Spanish Mountain Gold Project.

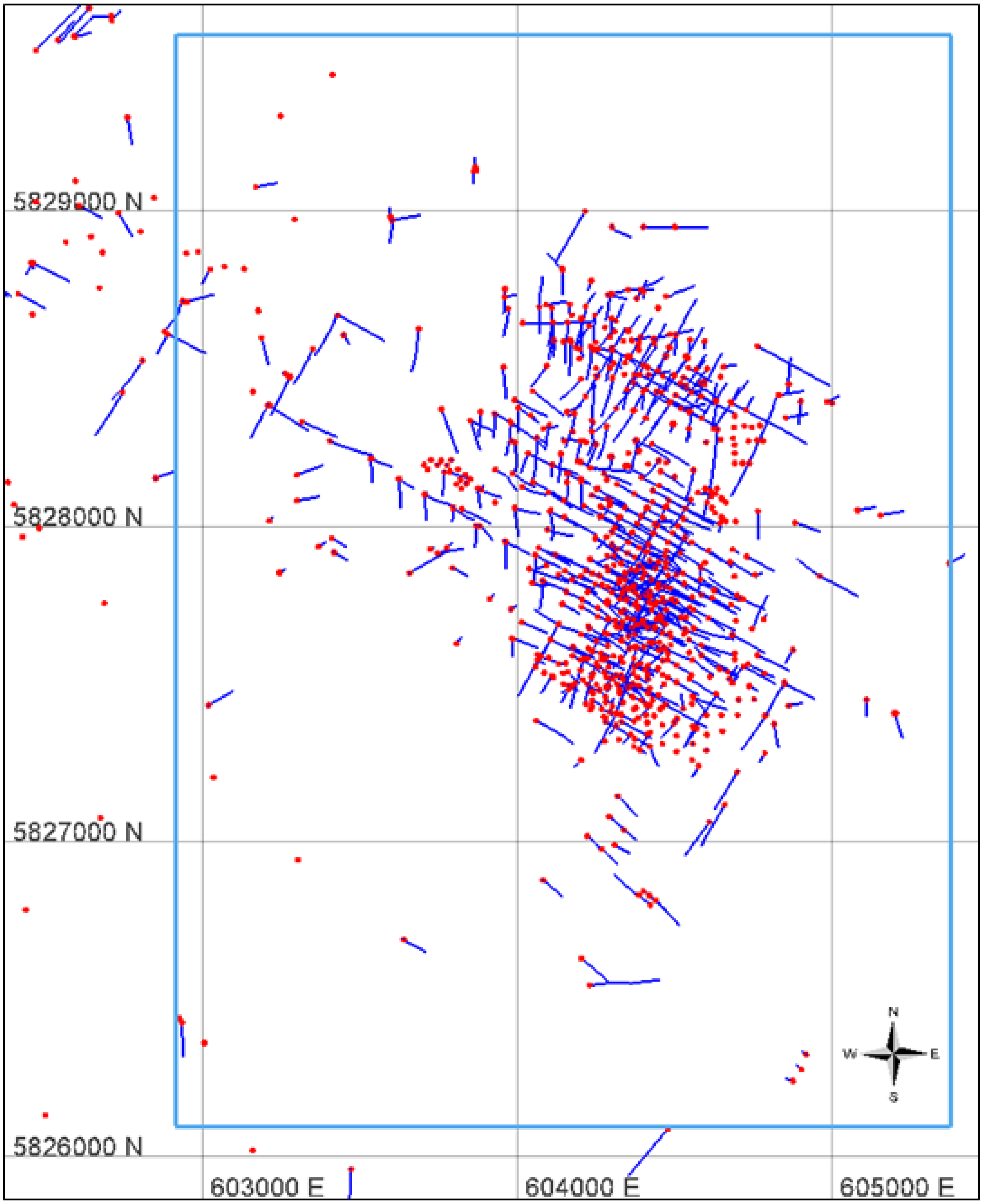


Figure 14-1 Drillhole Location Map Within the Block Model Limits Area (light blue) – All Drillhole Traces in Blue.

Table 14-5 Drillhole Spacing Statistics

	Mean (m)	
Rock Type*	**Mean (m)**	**Median (m)**
1-UARGS	20.8	18.1
2-UARGN	41.4	26.3
3-TUFFS	21.5	19.3
4-TUFFN	46.1	36.3
5-LARG	33.8	25.8
6-SLTSTS	22.3	18.2
7-SLTSTN	39.1	34.1
8-NZARG	33.0	24.7
Zones 1 to 8	30.1	23.3
9-OVB	36.8	26.8
All	30.0	23.2

**see Table 14-6 for rock type description*

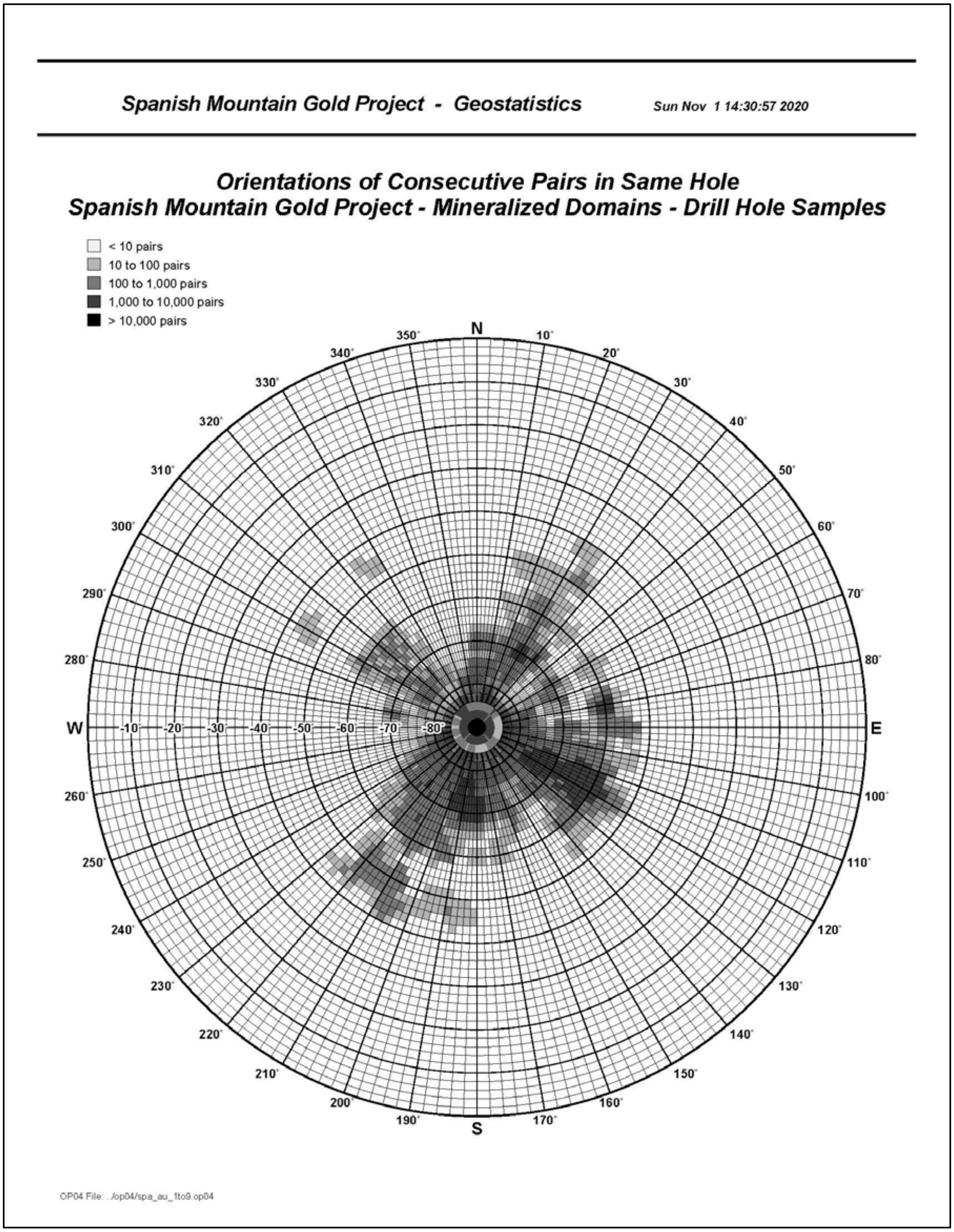


Figure 14-2 Stereonet of Drillhole Orientations

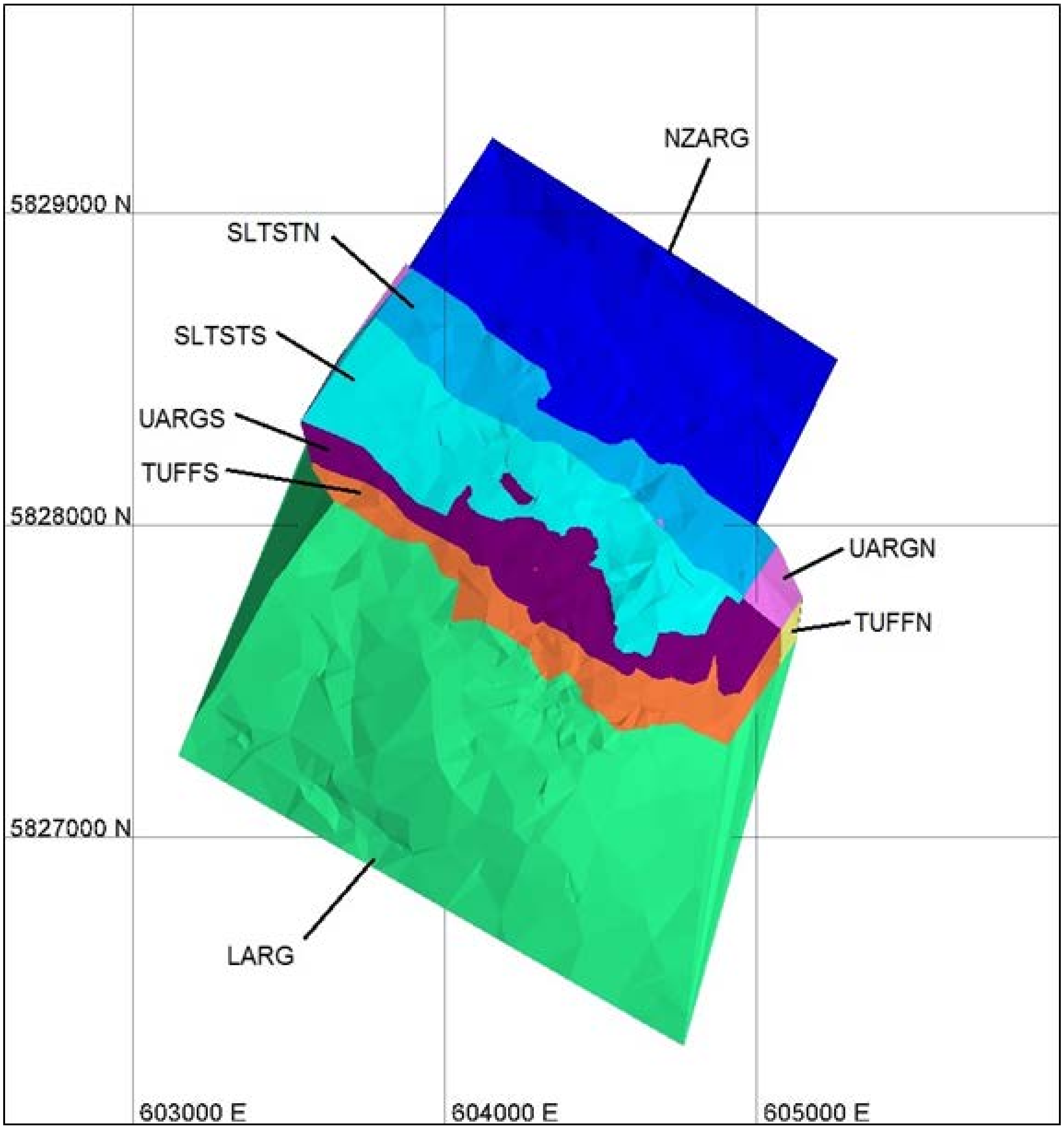
14.2 Geologic Modeling

The geologic model utilized for the Prefeasibility Study (PFS) remains the same as for the 2019 Preliminary Economic Assessment (PEA), as no new information, such as drillholes, were added since then. The geologic model was developed by the Spanish Mountain Gold team in 2017 and reviewed by Moose Mountain Technical Services for the 2019 PEA study. Marc Jutras is satisfied that the geological modelling reflects the current geological information and knowledge and is adequate for the estimation of Mineral Resources.

The main geologic controls on gold mineralization consist of preferred lithologic units identified within the area of interest. The main zone of gold mineralization is found within the Upper Argillite, Altered Siltstone, Tuff, Lower Argillite, and the North Zone Argillite. These lithologic units are oriented northwest-southeast at an azimuth of approximately 120° and dipping to the northeast. The Upper Argillic, Altered Siltstone, and Tuff units were subdivided into north and south domains to accommodate a change in dip within each unit. The south unit is generally dipping at a shallower angle of approximately 30° to the northeast, while the north unit is steeper dipping at an angle of approximately 60° to the northeast. A list of the lithologic units utilized for the estimation of the mineral resources is provided in Table 14-6. Representations of the lithologic solids from the geology model are displayed in Figure 14-3 in plan view, and in Figure 14-4 in perspective view. The triangulation of the overburden and topography surface is shown in Figure 14-5.

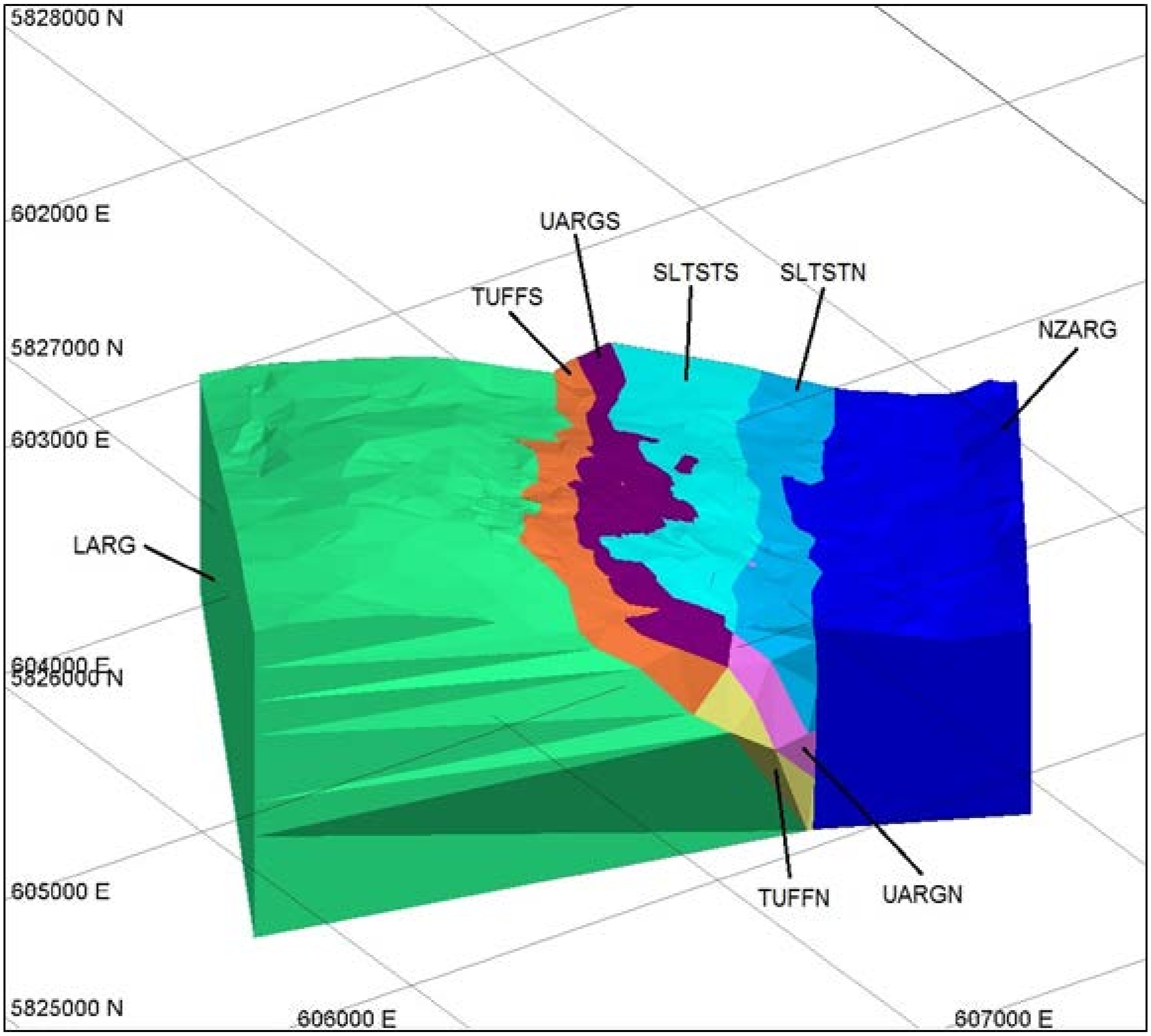
Table 14-6 List of Lithologic Units

Rock Code	Rock Type	Description	Volume m^3
1	UARGS	Upper Argillite - South	33,145,466,0
2	UARGN	Upper Argillite - North	41,384,136.4
3	TUFFS	Tuff - South	59,087,987.9
4	TUFFN	Tuff - North	83,694,228.3
5	LARG	Lower Argillite	1,811,770,373.8
6	SLTSTS	Siltstone - South	20,034,256.4
7	SLTSTN	Siltstone - North	66,455,863.2
8	NZARG	North Zone Argillite	433,029,447.4
9	OVB	Overburden	37,446,093.8



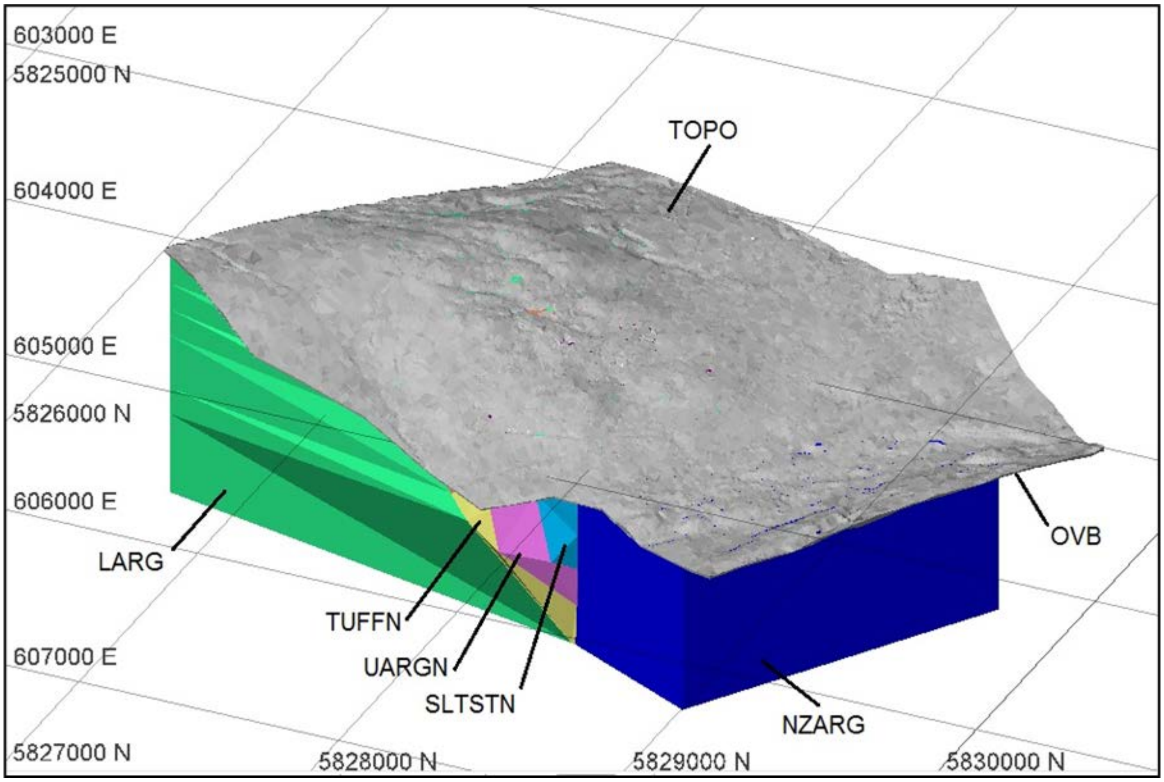
Source: Jutras, 2021

Figure 14-3 Geologic Model – Plan View



Source: Jutras, 2021

Figure 14-4 Geologic Model – Perspective View Looking Northwest



Source: Jutras, 2021

Figure 14-5 Geologic Model with Overburden and Topography – Perspective View Looking Southwest

14.3 Compositing

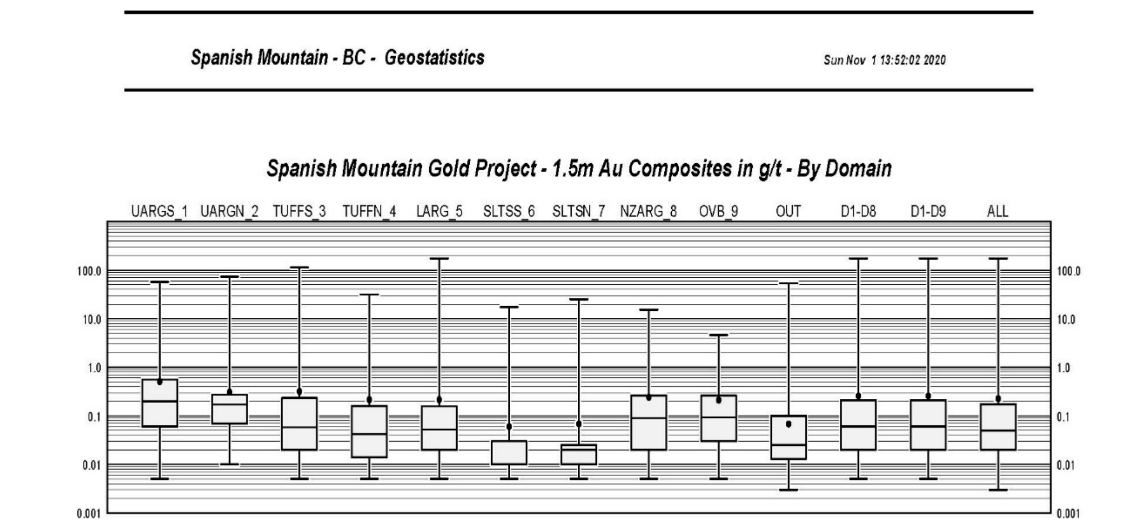
Statistics computed on the original sample lengths have shown that the most common sample length within the mineralized domains is 1.50m, with approximately 75% of the samples. To preserve the intrinsic variability of the original assays and to provide an equal support for these assays, the compositing of the original assays to regular 1.50m intervals was performed. In this process, the geologic units are considered as hard boundaries from which the compositing originates and ends. A dynamic compositing method was selected to avoid any shorter residual composites. In this approach, the length of the last and shorter assay (residual) is distributed to the other full-length assays, ensuring that all assays are of equal lengths. The composite length of 1.50m is within the composite length to block height guidelines of 1:2 to 1:5, with a ratio of 1:3.3 (1.5m/5.0m).

14.4 Exploratory Data Analysis (EDA)

A set of various statistical applications was utilized to provide a better understanding of the gold and silver grade populations within the various mineralized zones.

14.4.1 Univariate Statistics

Basic statistics were performed on the gold and silver grade composites of the lithologic units from the geologic model. Histograms and probability plots indicated that the gold and silver grade distributions resemble positively skewed lognormal populations. Basic statistics results are presented as boxplots per lithologic unit in Figure 14-6 and Figure 14-7, respectively for gold and silver. As seen in these figures, the gold grade populations are more heterogeneous with coefficients of variation (CV) greater than 3.0 in many of the units (six out of nine). This is most likely attributable to high gold grade values found in these units. The silver grades seem to be more homogeneous with higher coefficients of variation observed for two out of the nine lithologic units.



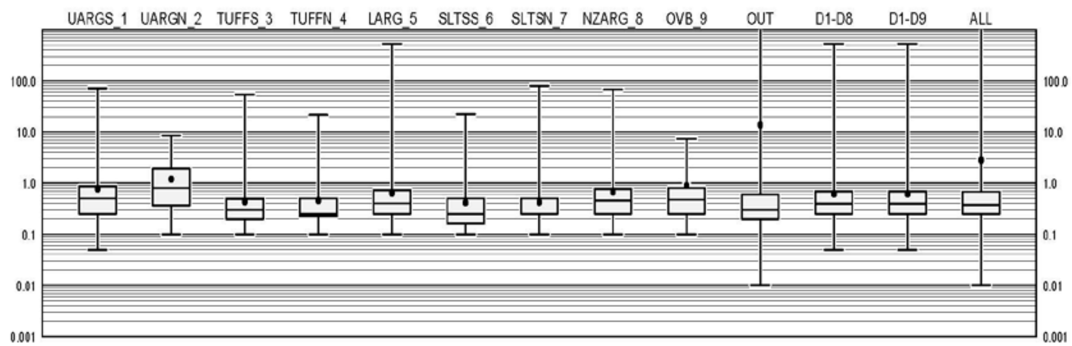
	UARGS_1	UARGN_2	TUFFS_3	TUFFN_4	LARG_5	SLTSS_6	SLTSN_7	NZARG_8	OVB_9	OUT	D1-D8	D1-D9	ALL	
Number of data	12253	3417	18658	4357	42831	3725	6647	18308	566	22330	110196	110762	133092	Number of data
Mean	0.506	0.309	0.318	0.216	0.215	0.06	0.068	0.24	0.211	0.068	0.258	0.258	0.226	Mean
Std. Dev.	1.174	1.5	1.599	0.984	1.342	0.49	0.501	0.517	0.317	0.418	1.214	1.212	1.121	Std. Dev.
Coef. of Var.	2.32	4.859	5.032	4.552	6.25	8.106	7.329	2.15	1.502	6.158	4.71	4.707	4.969	Coef. of Var.
Maximum	57.242	73.9	113.667	31.557	174.386	17.519	25.195	15.307	4.65	53.461	174.386	174.386	174.386	Maximum
Upper quartile	0.559	0.272	0.233	0.159	0.157	0.03	0.025	0.264	0.26	0.1	0.21	0.21	0.173	Upper quartile
Median	0.2	0.17	0.058	0.042	0.052	0.01	0.02	0.09	0.093	0.025	0.06	0.06	0.05	Median
Lower quartile	0.06	0.069	0.02	0.014	0.02	0.01	0.01	0.02	0.03	0.013	0.02	0.02	0.02	Lower quartile
Minimum	0.005	0.01	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.003	0.005	0.005	0.003	Minimum
Number of holes	320	73	397	84	443	158	99	169	376	223	802	815	1002	Number of holes

Variable: AU_GPT (weighted by length) from 0.0 to 100000.0
UARGS_1 file: ../op04/spa_au_1.op04
UARGN_2 file: ../op04/spa_au_2.op04
TUFFS_3 file: ../op04/spa_au_3.op04
TUFFN_4 file: ../op04/spa_au_4.op04
LARG_5 file: ../op04/spa_au_5.op04
SLTSS_6 file: ../op04/spa_au_6.op04
SLTSN_7 file: ../op04/spa_au_7.op04
NZARG_8 file: ../op04/spa_au_8.op04
OVB_9 file: ../op04/spa_au_9.op04
OUT file: ../op04/spa_au_out.op04
D1-D8 file: ../op04/spa_au_1to8.op04
D1-D9 file: ../op04/spa_au_1to9.op04
ALL file: ../op04/spa_au.op04

Figure 14-6 Basic Statistics of Gold Grade Composites

Spanish Mountain - BC - Geostatistics Sun Nov 1 13:54:58 2020

Spanish Mountain Gold Project - 1.5m Ag Composites in g/t - By Domain



	UARGS_1	UARGN_2	TUFFS_3	TUFFN_4	LARG_5	SLTSS_6	SLTSN_7	NZARG_8	OVB_9	OUT	D1-D8	D1-D9	ALL	
Number of data	11936	3417	18260	4357	41911	3719	6647	18308	524	22331	108555	109079	131410	Number of data
Mean	0.766	1.193	0.432	0.456	0.632	0.414	0.421	0.663	0.898	13.646	0.609	0.609	2.829	Mean
Std. Dev.	1.228	1.025	0.872	0.789	3.011	0.618	1.271	1.057	1.204	238.623	2.043	2.041	98.607	Std. Dev.
Coef. of Var.	1.603	0.859	2.019	1.731	4.76	1.493	3.019	1.595	1.34	17.486	3.357	3.35	34.853	Coef. of Var.
Maximum	71.389	8.555	53.432	21.548	527.633	22.258	79.9	67.559	7.32	18135.0	527.633	527.633	18135.0	Maximum
Upper quartile	0.861	1.897	0.498	0.5	0.734	0.5	0.499	0.773	0.8	0.6	0.689	0.692	0.68	Upper quartile
Median	0.5	0.8	0.3	0.25	0.401	0.25	0.25	0.453	0.474	0.299	0.394	0.395	0.374	Median
Lower quartile	0.25	0.368	0.2	0.225	0.25	0.163	0.25	0.25	0.25	0.2	0.25	0.25	0.25	Lower quartile
Minimum	0.05	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.01	0.05	0.05	0.01	Minimum
Number of holes	311	73	389	84	433	157	99	169	364	223	784	797	984	Number of holes

Variable: AG_GPT (weighted by length) from 0.01 to 100000.0
UARGS_1 file: ../op04/spa_ag_1.op04
UARGN_2 file: ../op04/spa_ag_2.op04
TUFFS_3 file: ../op04/spa_ag_3.op04
TUFFN_4 file: ../op04/spa_ag_4.op04
LARG_5 file: ../op04/spa_ag_5.op04
SLTSS_6 file: ../op04/spa_ag_6.op04
SLTSN_7 file: ../op04/spa_ag_7.op04
NZARG_8 file: ../op04/spa_ag_8.op04
OVB_9 file: ../op04/spa_ag_9.op04
OUT file: ../op04/spa_ag_out.op04
D1-D8 file: ../op04/spa_ag_1to8.op04
D1-D9 file: ../op04/spa_ag_1to9.op04
ALL file: ../op04/spa_ag.op04

Figure 14-7 Basic Statistics of Silver Grade Composites

14.4.2 Capping of High-Grade Outliers

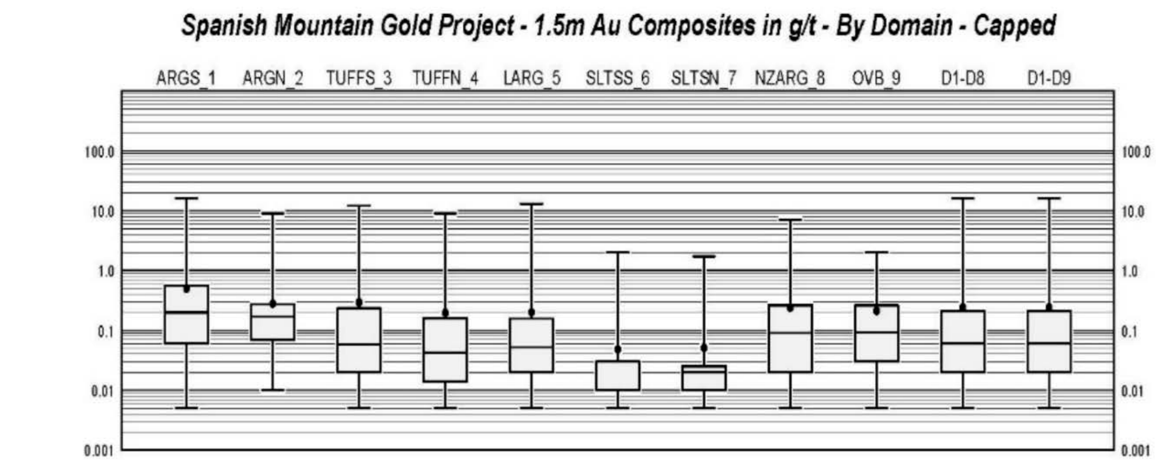
It is common practice to statistically examine the higher grades within a population and to trim them to a lower grade value based on the results from specific statistical utilities. This procedure is performed on high-grade values that are considered outliers and that cannot be related to any geologic feature. In the case for the Spanish Mountain Gold Project, the higher gold and silver grades were examined with three different tools: the probability plot, decile analysis, and cutting statistics. The usage of various investigating methods allows for a selection of the capping threshold in a more objective and justified manner. For the probability plot method, the capping value is chosen at the location where higher grades depart from the main distribution. For the decile analysis, the capping value is chosen as the maximum grade of the decile containing less than an average of 10% of metal. For the cutting statistics, the selection of the capping value is identified at the cut-off grade where there is no correlation between the grades above this cut-off. The resulting compilation of the capping thresholds is listed in Table 14-7. One of the objectives of the capping strategy is to have less than 10% of the metal affected by the capping process. This was achieved in most of the cases, except for the gold capping of the siltstone units. For these units, the few higher gold grades within a predominantly low gold grade population, contain a greater proportion of the metal content.

Table 14-7 List of Capping Thresholds of Higher Gold and Silver Grade Outliers

Au Capping Thresholds			
Rock Type	**Capping Threshold g/t**	**% Metal Affected**	**Number of Comps Capped**
1-UARGS	16.0	2	6
2-UARGN	9.0	9	4
3-TUFFS	12.0	7	35
4-TUFFN	9.0	8	9
5-LARG	13.0	5	31
6-SLTSTS	2.0	20	8
7-SLTSTN	1.7	24	33
8-NZARG	7.0	1	11
9-OVB	2.0	4	3
Ag Capping Thresholds			
1-UARGS	15.0	1	9
2-UARGN	6.0	1	3
3-TUFFS	15.0	1	13
4-TUFFN	14.0	1	3
5-LARG	18.0	4	15
6-SLTSTS	8.0	2	2
7-SLTSTN	15.0	4	4
8-NZARG	27.0	1	3
9-OVB	5.0	2	7

Basic statistics were re-computed with the gold and silver grades capped to the thresholds listed in Table 14-7. Boxplots of Figure 14-8 to Figure 14-9 display the basic statistics resulting from the capping of the higher gold and silver grade outliers. It can be observed from those Figures that the coefficients of variation are in general below or close to 3.0 for the different gold grade populations. The populations of the capped silver grades show more homogenous distributions with coefficients of variation below 2.0. The effect of the capping of higher-grade outliers has slightly reduced the overall mean gold grade by 5.8% and the overall mean silver grade by 2.1%.

Because of the generally low coefficients of variation observed for the gold and silver grade populations, it was concluded that the usage of ordinary kriging with capped gold and silver composites is a well-suited estimation technique in this case.

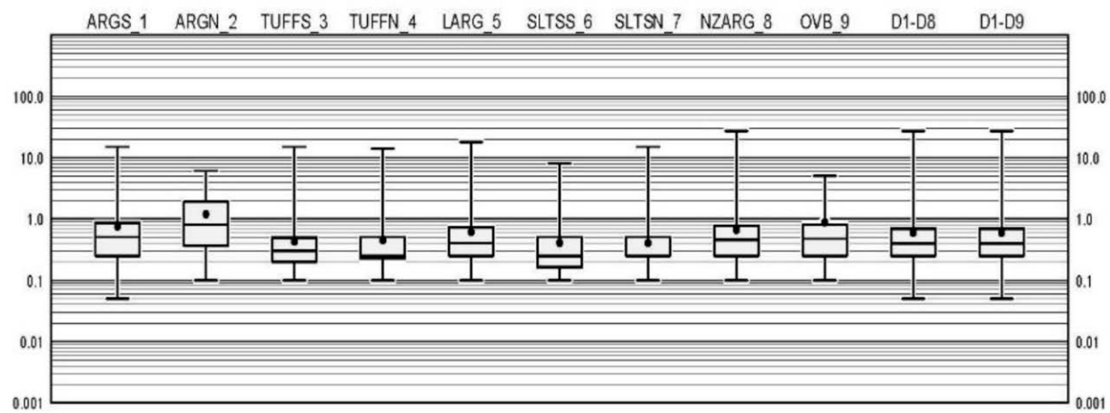


	ARGS_1	ARGN_2	TUFFS_3	TUFFN_4	LARG_5	SLTSS_6	SLTSN_7	NZARG_8	OVB_9	D1-D8	D1-D9	
Number of data	12253	3417	18658	4357	42831	3725	6647	18308	566	110196	110762	Number of data
Mean	0.497	0.278	0.293	0.197	0.2	0.048	0.051	0.237	0.211	0.243	0.243	Mean
Std. Dev.	0.906	0.541	0.891	0.601	0.613	0.144	0.156	0.45	0.317	0.665	0.665	Std. Dev.
Coef. of Var.	1.825	1.948	3.044	3.053	3.062	3.016	3.061	1.897	1.502	2.739	2.736	Coef. of Var.
Maximum	16.0	9.0	12.0	9.0	13.0	2.0	1.7	7.0	2.0	16.0	16.0	Maximum
Upper quartile	0.559	0.272	0.233	0.159	0.157	0.03	0.025	0.264	0.26	0.21	0.21	Upper quartile
Median	0.2	0.17	0.058	0.042	0.052	0.01	0.02	0.09	0.093	0.06	0.06	Median
Lower quartile	0.06	0.069	0.02	0.014	0.02	0.01	0.01	0.02	0.03	0.02	0.02	Lower quartile
Minimum	0.005	0.01	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	Minimum
Number of holes	320	73	397	84	443	158	99	169	376	1743	2119	Number of holes

Variable: AU_GPT (weighted by length) from 0.0 to 100000.0
ARGS_1 file: ./cp04/spa_au_1.cp04.cut
ARGN_2 file: ./cp04/spa_au_2.cp04.cut
TUFFS_3 file: ./cp04/spa_au_3.cp04.cut
TUFFN_4 file: ./cp04/spa_au_4.cp04.cut
LARG_5 file: ./cp04/spa_au_5.cp04.cut
SLTSS_6 file: ./cp04/spa_au_6.cp04.cut
SLTSN_7 file: ./cp04/spa_au_7.cp04.cut
NZARG_8 file: ./cp04/spa_au_8.cp04.cut
OVB_9 file: ./cp04/spa_au_9.cp04.cut
D1-D8 file: ./cp04/spa_au_1to8.cp04.cut
D1-D9 file: ./cp04/spa_au_1to9.cp04.cut

Figure 14-8 Basic Statistics of Capped Gold Grade Composites

Spanish Mountain Gold Project - 1.5m Ag Composites in g/t - By Domain - Capped



	ARGS_1	ARGN_2	TUFFS_3	TUFFN_4	LARG_5	SLTSS_6	SLTSN_7	NZARG_8	OVB_9	D1-D8	D1-D9	
Number of data	11936	3417	18260	4357	41911	3719	6647	18308	524	108555	109079	Number of data
Mean	0.754	1.192	0.426	0.452	0.61	0.408	0.404	0.659	0.874	0.596	0.596	Mean
Std. Dev.	0.876	1.019	0.658	0.707	0.752	0.479	0.586	0.913	1.099	0.788	0.789	Std. Dev.
Coef. of Var.	1.162	0.855	1.546	1.565	1.233	1.174	1.453	1.385	1.258	1.324	1.324	Coef. of Var.
Maximum	15.0	6.0	15.0	14.0	18.0	8.0	15.0	27.0	5.0	27.0	27.0	Maximum
Upper quartile	0.861	1.897	0.498	0.5	0.734	0.5	0.499	0.773	0.8	0.689	0.692	Upper quartile
Median	0.5	0.8	0.3	0.25	0.401	0.25	0.25	0.453	0.474	0.394	0.395	Median
Lower quartile	0.25	0.368	0.2	0.225	0.25	0.163	0.25	0.25	0.25	0.25	0.25	Lower quartile
Minimum	0.05	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.05	0.05	Minimum
Number of holes	311	73	389	84	433	157	99	169	364	1715	2079	Number of holes

Variable: AG_GPT (weighted by length) from 0.0 to 100000.0
ARGS_1 file: ./cp04/spa_ag_1.cp04.cut
ARGN_2 file: ./cp04/spa_ag_2.cp04.cut
TUFFS_3 file: ./cp04/spa_ag_3.cp04.cut
TUFFN_4 file: ./cp04/spa_ag_4.cp04.cut
LARG_5 file: ./cp04/spa_ag_5.cp04.cut
SLTSS_6 file: ./cp04/spa_ag_6.cp04.cut
SLTSN_7 file: ./cp04/spa_ag_7.cp04.cut
NZARG_8 file: ./cp04/spa_ag_8.cp04.cut
OVB_9 file: ./cp04/spa_ag_9.cp04.cut
D1-D8 file: ./cp04/spa_ag_1to8.cp04.cut
D1-D9 file: ./cp04/spa_ag_1to9.cp04.cut

Figure 14-9 Basic Statistics of Capped Silver Grade Composites

14.4.3 Declustering

In general, there is a tendency to drill more holes in higher grade areas than in lower grade areas when delimiting an orebody. As a result, the higher-grade portion of a deposit will be overly represented and would translate into a bias towards the higher grades when calculating statistical parameters of the population. Thus, a declustering method is utilized to generate a more representative set of statistical results within the zone of interest. In this case, a polygonal declustering technique was applied to the gold and silver composites within the mineralized domains of the Spanish Mountain Gold Project. This approach consists of assigning the volume of a polygon, defined by the halfway distance between a sample and its surrounding neighbours, as a weight for each sample within the mineralized domains. Therefore, a sample that is isolated will have a larger weight than a sample located in a densely drilled area.

Comparisons of average capped and declustered grades with the capped and un-declustered grades show some clustering overall with decreases in average gold declustered grades by 32.9% and decreases in average silver declustered grades by 10.9%.

The average grade from the declustered statistics provides an excellent comparison with the average grade of the interpolated blocks, to assess any overall bias of the estimates.

14.5 Variography

A variographic analysis was carried out on the gold and silver grade composites within the different mineralized domains. The objective of this analysis was to spatially establish the preferred directions of gold and silver grade continuity. In turn, the variograms modeled along those directions would be later utilized to select and weigh the composites during the block grade interpolation process. For this exercise, all experimental variograms were of the type relative lag pairwise, which is considered robust for the assessment of gold and silver grade continuity.

Variogram maps were first calculated to examine general gold and silver grade continuities in the XY, XZ, and YZ planes. The next step undertaken was to compute omni-directional variograms and down-hole variograms. The omni-directional variograms are calculated without any directional restrictions and provide a good assessment of the sill of the variogram. As for the down-hole variogram, it is calculated with the composites of each hole along the trace of the hole. The objective of these calculations is to provide information about the short scale structure of the variogram, as the composites are more closely spaced down the hole. Thus, the modeling of the nugget effect is usually better derived from the down-hole variograms.

Directional variograms were then computed to identify more specifically the three main directions of continuity. A first set of variograms were produced in the horizontal plane at increments of 10 degrees. In the same way, a second set of variograms were computed at 10° increments in the vertical plane of the horizontal direction of continuity (plunge direction). A final set of variograms at 10° increments were calculated in the vertical plane perpendicular to the horizontal direction of continuity (dip direction). The final variograms were then modeled with a two-structure spherical variogram, and resulting parameters presented in Table 14-8 for gold, and Table 14-9 for silver. The experimental and modeled variograms for gold are presented in Appendix A.

The directions of gold grade continuity are in general agreement with the orientation of the mineralized zone, with best directions of continuity trending west-northwest/east-southeast at an azimuth of 120° and down-dip at angles varying from 25° to 40° northeast for the southern domains and at angles varying from 55° to 65° northeast for the northern domains. The ranges of grade continuity along the principal direction (strike) vary from 49m to 73m for gold, and from 55m to 84m for silver. Along the minor direction (dip), the ranges of continuity vary from 36m to 82m for gold and from 47m to 80m for silver. Finally, along the vertical direction (across strike and dip), the ranges of continuity vary from 16m to 74m for gold, and from 19m to 43m for silver. The modeled variograms have relatively low nugget effects with values varying from 8% to 18% of the sill for gold, and from 6% to 14% of the sill for silver.

The experimental variograms are considered of good quality overall, most likely due to the abundance of available data and sufficient drill spacing.

Table 14-8 Modeled Variogram Parameters for Gold Composites

Parameters	1 – UARGS			2 – UARGN			3 – TUFFS		
	Principal	Minor	Vertical	Principal	Minor	Vertical	Principal	Minor	Vertical
Azimuth*	120°	210°	210°	120°	210°	210°	120°	210°	210°
Dip**	0°	30°	-60°	0°	65°	-25°	0°	30°	-60°
Nugget Effect C_0	0.213			0.130			0.292		
1st Structure C_1	0.921			0.574			1.053		
2nd Structure C_2	0.471			0.345			0.324		
1st Range A_1	13.5m	12.4m	10.3m	40.4m	39.3m	12.9m	10.3m	6.4m	10.3m
2nd Range A_2	63.9m	65.0m	40.3m	64.0m	65.1m	24.7m	48.9m	61.2m	43.6m
Parameters	4 – TUFFN			5 – LARG			6 - SLTSTS		
	Principal	Minor	Vertical	Principal	Minor	Vertical	Principal	Minor	Vertical
Azimuth*	120°	210°	210°	110°	200°	200°	120°	210°	210°
Dip**	0°	60°	-30°	0°	40°	-50°	0°	25°	-65°
Nugget Effect C_0	0.234			0.213			0.131		
1st Structure C_1	0.930			0.884			0.498		
2nd Structure C_2	0.499			0.409			0.272		
1st Range A_1	12.4m	18.9m	33.9m	10.3m	7.1m	21.0m	11.4m	7.1m	11.4m
2nd Range A_2	60.7m	59.6m	43.5m	71.5m	72.5m	73.6m	51.2m	36.1m	22.1m
Parameters	7 - SLTSTN			8 - NZARG			9 - OVB		
	Principal	Minor	Vertical	Principal	Minor	Vertical	Principal	Minor	Vertical
Azimuth*	120°	210°	210°	120°	210°	210°	85°	175°	175°
Dip**	0°	55°	-35°	0°	50°	-40°	0°	0°	-90°
Nugget Effect C_0	0.101			0.117			0.124		
1st Structure C_1	0.450			0.987			0.524		
2nd Structure C_2	0.232			0.409			0.887		
1st Range A_1	58.6m	53.2m	11.4m	16.7m	12.4m	11.4m	39.5m	33.0m	10.3m
2nd Range A_2	61.8m	82.2m	38.2m	72.6m	53.2m	45.7m	65.4m	58.9m	15.7m

**positive clockwise from north*
***negative below horizontal*

Table 14-9 Modeled Variogram Parameters for Silver Composites

Parameters	1 – UARGS			2 – UARGN			3 – TUFFS		
	Principal	Minor	Vertical	Principal	Minor	Vertical	Principal	Minor	Vertical
Azimuth*	120°	210°	210°	120°	210°	210°	120°	210°	210°
Dip**	0°	30°	-60°	0°	65°	-25°	0°	30°	-60°
Nugget Effect C_0	0.051			0.054			0.086		
1st Structure C_1	0.398			0.311			0.409		
2nd Structure C_2	0.371			0.485			0.162		
1st Range A_1	7.1m	7.1m	7.1m	21.0m	33.9m	16.7m	8.1m	8.1m	18.9m
2nd Range A_2	73.6m	50.0m	25.3m	81.2m	53.3m	25.3m	68.3m	66.1m	42.5m
Parameters	**4 – TUFFN**			**5 – LARG**			**6 - SLTSTS**		
	Principal	**Minor**	**Vertical**	**Principal**	**Minor**	**Vertical**	**Principal**	**Minor**	**Vertical**
Azimuth*	120°	210°	210°	110°	200°	200°	120°	210°	210°
Dip**	0°	60°	-30°	0°	40°	-50°	0°	25°	-65°
Nugget Effect C_0	0.087			0.087			0.102		
1st Structure C_1	0.489			0.400			0.420		
2nd Structure C_2	0.124			0.300			0.214		
1st Range A_1	12.4m	52.1m	21.0m	15.6m	7.1m	15.6m	34.0m	26.4m	18.9m
2nd Range A_2	55.3m	71.4m	34.9m	70.4m	57.5m	42.5m	74.9m	46.9m	31.8m
Parameters	**7 - SLTSTN**			**8 - NZARG**			**9 - OVB**		
	Principal	**Minor**	**Vertical**	**Principal**	**Minor**	**Vertical**	**Principal**	**Minor**	**Vertical**
Azimuth*	115°	205°	205°	120°	210°	210°	110°	200°	200°
Dip**	0°	45°	-45°	0°	50°	-40°	0°	5°	-85°
Nugget Effect C_0	0.069			0.054			0.082		
1st Structure C_1	0.210			0.419			0.270		
2nd Structure C_2	0.204			0.201			0.414		
1st Range A_1	28.5m	30.3m	6.0m	17.8m	28.5m	21.0m	29.5m	15.6m	12.4m
2nd Range A_2	73.6m	80.1m	32.8m	84.4m	57.5m	39.3m	56.2m	59.4m	18.8m

**positive clockwise from north*
***negative below horizontal*

14.6 Grade Estimation

The estimation of grades into a block model was carried out with the ordinary kriging technique on capped gold and silver 1.5m composites. The estimation strategy and parameters were tailored to account for the various geometrical, geological, and geostatistical characteristics previously identified. The block grid definition is presented in Table 14-10. It should be noted that the origin of the block model corresponds to the lower left corner, the point of origin being the exterior edges of the first block. A block size of 15m (easting) x 15m (northing) x 5m (elevation) was selected to better reflect the orebody's geometrical configuration and anticipated open pit production rate. The block model is orthogonal with no rotation applied to it.

Table 14-10 Block Grid Definition

Block Model					
Coordinates	Origin m	Rotation	Distance m	Block Size m	Number of Blocks
Easting (X)	602,915.0	0°	2,460.0	15.0	164
Northing (Y)	5,826,095.0		3,465.0	15.0	231
Elevation(Z)	245.0		1,205.0	5.0	241
Number of Blocks		327,040,000			

The database of 1.5m capped gold and silver grade composites was utilized as input for the grade interpolation process.

The size and orientation of the search ellipsoid for the estimation of gold grades was based on the variogram parameters modeled for gold and, similarly for the estimation of silver grades, the size and orientation of the search ellipsoid was based on the variogram parameters modeled for silver. A minimum of two samples and maximum of eight samples was selected for the block estimation of gold grades and, a minimum of two samples and maximum of twelve samples was selected for the estimation of silver grades. No other restrictions, such as a minimum number of informed octants, a minimum number of holes, a maximum number of samples per hole, etc., were applied to the estimation process. A summary of the estimation parameters is presented in Table 14-11.

Table 14-11 Estimation Parameters for Gold

Estimation Parameters – Gold Grade – Spanish Mountain Gold Project								
Rock Code	**minimum # of samples**	**maximum # of samples**	**search ellipsoid – long axis - azimuth/dip**	**search ellipsoid – long axis - size**	**search ellipsoid – short axis - azimuth/dip**	**search ellipsoid – short axis - size**	**search ellipsoid – vertical axis - azimuth/dip**	**search ellipsoid – vertical axis - size**
1	2	8	120°/0°	64.0m	210°/30°	65.0m	210°/-60°	40.0m
2	2	8	120°/0°	64.0m	210°/65°	65.0m	210°/-25°	25.0m
3	2	8	120°/0°	49.0m	210°/30°	61.0m	210°/-60°	44.0m
4	2	8	120°/0°	61.0m	210°/60°	60.0m	210°/-30°	44.0m
5	2	8	110°/0°	72.0m	200°/40°	73.0m	200°/-50°	74.0m
6	2	8	120°/0°	51.0m	210°/25°	36.0m	210°/-65°	22.0m
7	2	8	120°/0°	62.0m	210°/55°	82.0m	210°/-35°	38.0m
8	2	8	120°/0°	73.0m	210°/50°	53.0m	210°/-40°	46.0m
9	2	8	85°/0°	65.0m	175°/0°	59.0m	175°/-90°	16.0m
Estimation Parameters – Silver Grade – Spanish Mountain Gold Project								
1	2	12	120°/0°	74.0m	210°/30°	50.0m	210°/-60°	25.0m
2	2	12	120°/0°	81.0m	210°/65°	53.0m	210°/-25°	25.0m
3	2	12	120°/0°	68.0m	210°/30°	66.0m	210°/-60°	43.0m
4	2	12	120°/0°	55.0m	210°/60°	71.0m	210°/-30°	35.0m
5	2	12	110°/0°	70.0m	200°/40°	58.0m	200°/-50°	43.0m
6	2	12	120°/0°	75.0m	210°/25°	47.0m	210°/-65°	32.0m
7	2	12	115°/0°	74.0m	205°/45°	80.0m	205°/-45°	33.0m
8	2	12	120°/0°	84.0m	210°/50°	58.0m	210°/-40°	39.0m
9	2	12	110°/0°	56.0m	200°/5°	59.0m	200°/-85°	19.0m

The grade estimation process consisted of a two-pass approach with the parameters of the first pass as presented in Table 14-11. The estimation parameters of the second pass are the same except for an enlarged search ellipsoid by two times the dimensions from the first pass. In this case, priority was given to estimates from the first pass, followed by estimates from the second pass. Only blocks within the mineralized domains were estimated.

The boundary conditions between the different mineralized domains were examined with contact plots. A summary of the boundary conditions is presented in Table 14-12 for gold and silver.

Table 14-12 Boundary Conditions

Bordering Domains	Boundary Condition	
	Gold	Silver
1-UARGS / 2-UARGN	soft	soft
1-UARGS / 3-TUFFS	hard	hard
1-UARGS / 6-SLTSTS	hard	hard
1-UARGS / 9-OVB	hard	hard
2-UARGN / 8-NZARG	soft	soft
3-TUFFS / 4-TUFFN	hard	soft
3-TUFFS / 5-LARG	hard	soft
3-TUFFS / 9-OVB	hard	hard
4-TUFFN / 5-LARG	soft	soft
4-TUFFN / 8-NZARG	hard	hard
5-LARG / 9-OVB	hard	soft
6-SLTSTS / 7-SLTSTN	soft	soft
6-SLTSTS / 9-OVB	soft	hard
7-SLTSTN / 8-NZARG	hard	hard
7-SLTSTN / 9-OVB	soft	hard
8-NZARG / 9-OVB	hard	hard

14.7 Validation of Grade Estimates

Validation tests were carried out on the estimates to examine the possible presence of a bias and to quantify the level of smoothing/variability.

14.7.1 Visual Inspection

A visual inspection of the gold and silver estimates with the drillhole grades on plans, east-west and north-south cross-sections was performed as a first check of the estimates. Observations from stepping through the estimates along the different planes indicated that there was overall a good agreement between the drillhole grades and the estimates. The orientations of the estimated grades were also according to the projection angles defined by the search ellipsoid. Examples of cross-sections and level plans for gold grade estimates of the different mineralized zones are presented in Figure 14-10 to Figure 14-12. Examples of cross-sections and level plans for silver grade estimates of the different mineralized zones are presented in Figure 14-13 to Figure 14-15.

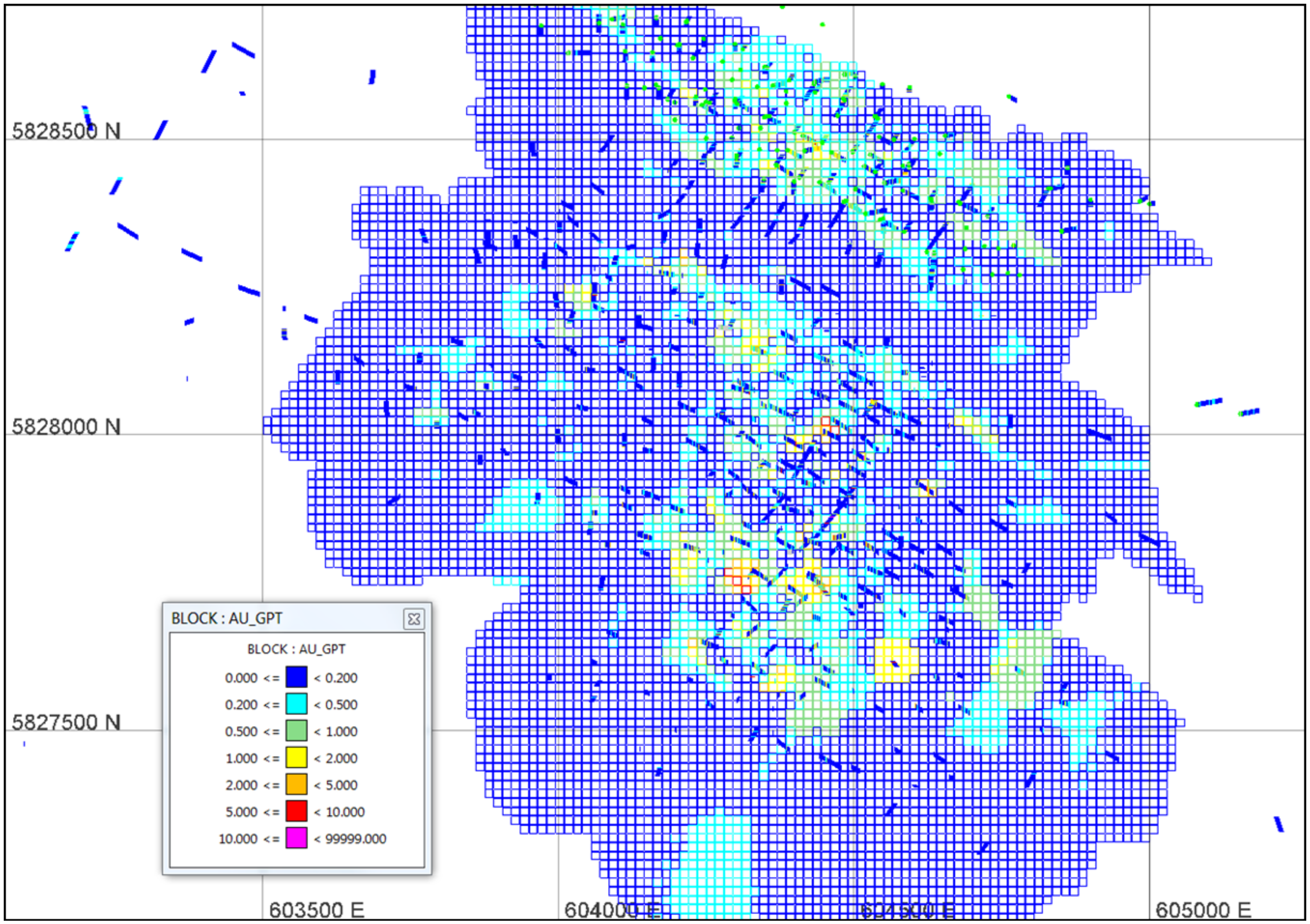


Figure 14-10 Gold Block Grade Estimates and Drillhole Grades – Level 925 El

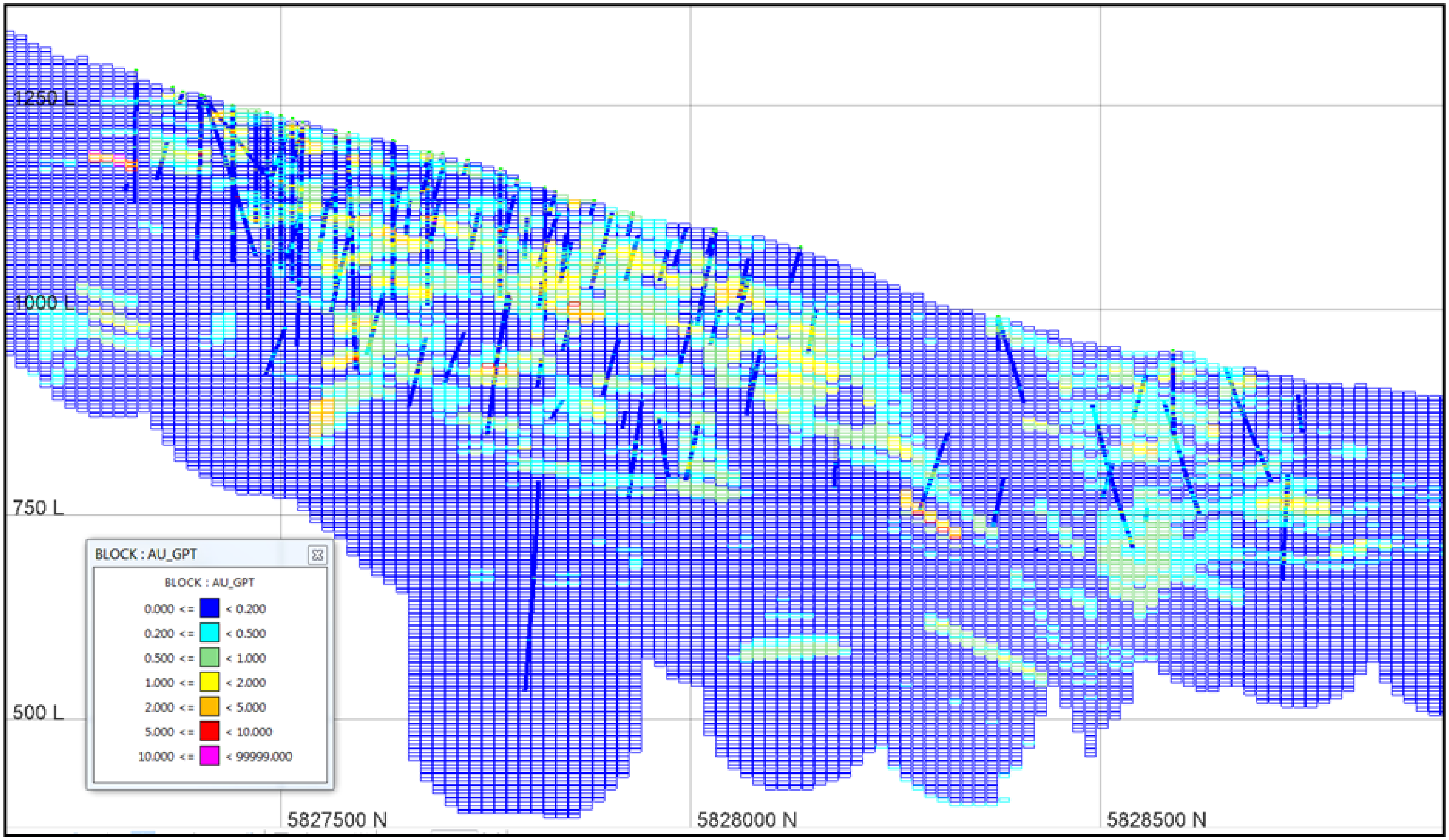


Figure 14-11 Gold Block Grade Estimates and Drillhole Grades – Section 604325E

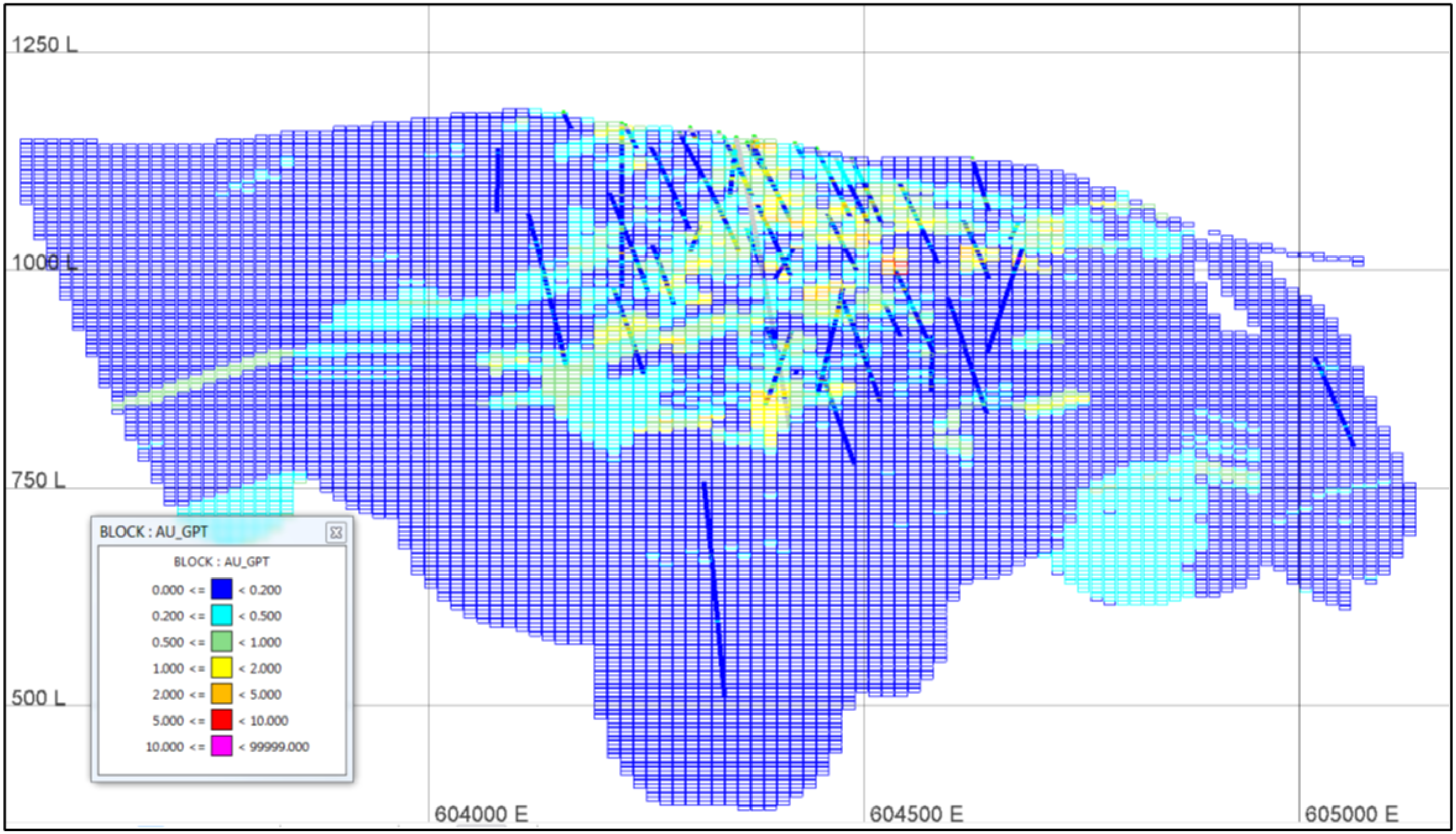


Figure 14-12 Gold Block Grade Estimates and Drillhole Grades – Section 5827800N

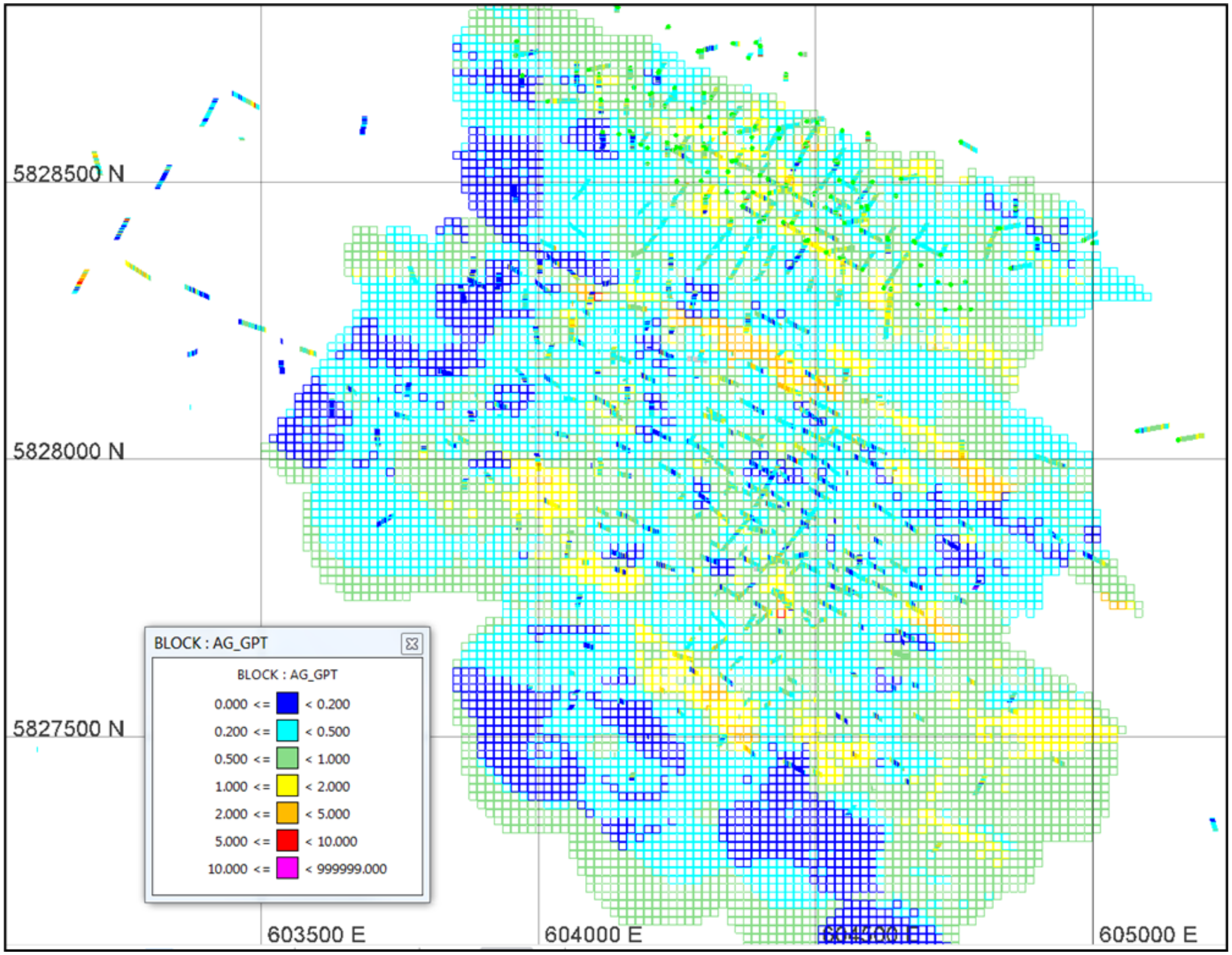


Figure 14-13 Silver Block Grade Estimates and Drillhole Grades – Level 925 El

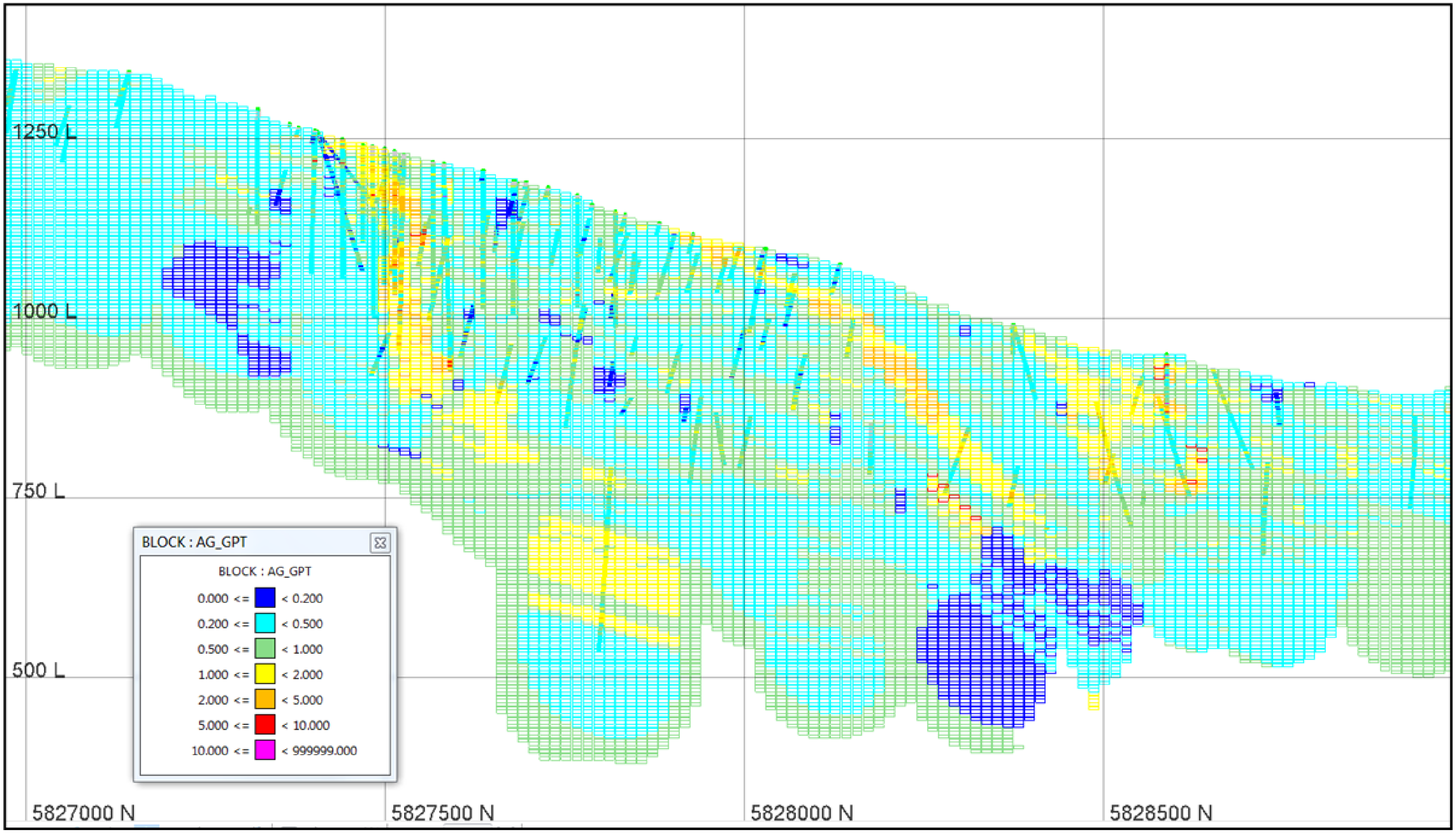


Figure 14-14 Silver Block Grade Estimates and Drillhole Grades – Section 604325E

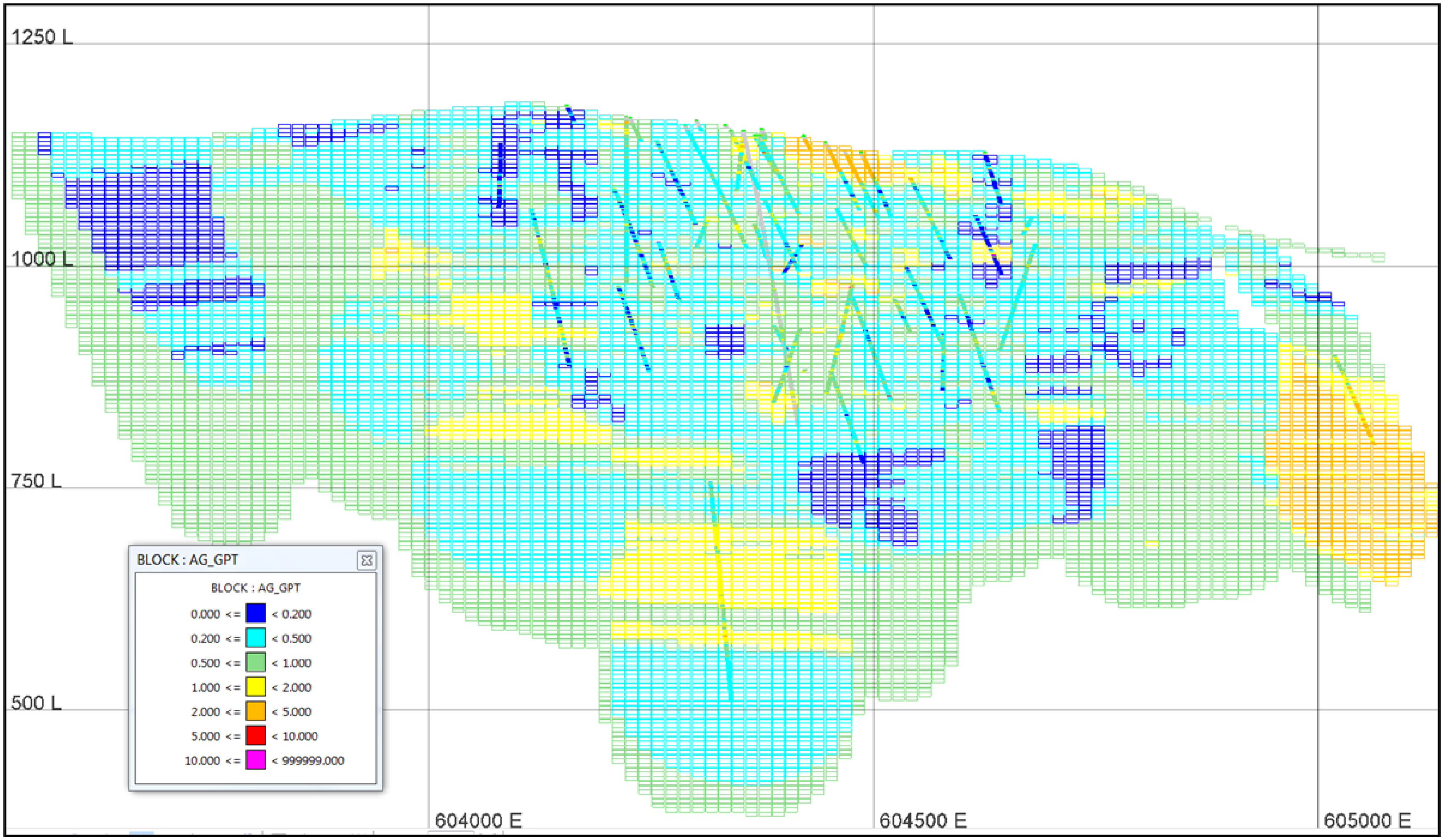


Figure 14-15 Silver Block Grade Estimates and Drillhole Grades – Section 5827800N

14.7.2 Global Bias Test

The comparison of the average grades from the declustered composites and the estimated block grades examines the possibility of a global bias of the estimates. As a guideline, a difference between the average grades of more than ± 10% would indicate a significant over- or under-estimation of the block grades and the possible presence of a bias. It would be a sign of difficulties encountered in the estimation process and would require further investigation.

Results of the average grade comparisons are presented in Table 14-13 for gold and silver.

Table 14-13 Average Gold and Silver Grade Comparison – Polygonal-Declustered Composites with Block Estimates

Gold		
Statistics	**Declustered Composites**	**Block Estimates**
Average Gold Grade g/t	0.163	0.163
Difference		0.0%
Silver		
Statistics	**Declustered Composites**	**Block Estimates**
Average Gold Grade g/t	0.531	0.524
Difference		-1.3%

As seen in Table 14-13, the average gold grades between the declustered composites and the block estimates are within the limits of the tolerance levels of acceptability. It can be concluded that no significant global bias is present in the gold and silver grade estimates.

14.7.3 Local Bias Test

A comparison of the grade from composites within a block with the estimated grade of that block provides an assessment of the estimation process close to measured data. Pairing of these grades on a scatterplot gives a statistical valuation of the estimates. It is anticipated that the estimated block grades should be like the composited grades within the block, however without being of the same value. Thus, a high correlation coefficient will indicate satisfactory results in the interpolation process, while a medium to low correlation coefficient will be indicative of larger differences in the estimates and would suggest a further review of the interpolation process. Results from the pairing of composited and estimated grades within blocks pierced by a drillhole are presented in Table 14-14 for gold and silver.

Table 14-14 Gold and Silver Grade Comparison for Blocks Pierced by a Drillhole – Paired Composite Grades with Block Grade Estimates

Gold		
Data	**Average Gold Grade g/t**	**Difference**
Composites	0.247	-0.4%
Block Estimates	0.246	
Correlation Coefficient	0.681	
Silver		
Data	**Average Silver Grade g/t**	**Difference**
Composites	0.593	0.2%
Block Estimates	0.594	
Correlation Coefficient	0.736	

As seen in Table 14-14 for gold and silver, the block grade estimates are very similar to the composite grades within blocks pierced by a drillhole, with high correlation coefficients, indicating satisfactory results from the estimation process.

14.7.4 Grade Profile Reproducibility

The comparison of the grade profiles of the declustered composites with that of the estimates allows for a visual verification of an over- or under-estimation of the block estimates at the global and local scales. A qualitative assessment of the smoothing/variability of the estimates can also be observed from the plots. The output consists of three graphs displaying the average grade according to each of the coordinate axes (east, north, elevation). The ideal result is a grade profile from the estimates that follows that of the declustered composites along the three coordinate axes, in a way that the estimates have lower high-grade peaks than the composites, and higher low-grade peaks than the composites. A smoother grade profile for the estimates, from low to high grade areas, is also anticipated to reflect that these grades represent larger volumes than the composites.

Gold grade profiles are presented in Figure 14-16 and Figure 14-17 for gold and in Figure 14-18 and Figure 14-19 for silver. From these plots the grade profiles of the declustered composites are well reproduced by those of the block estimates and consequently that no global or local bias is observed. As anticipated, some smoothing of the block estimates can be seen in the profiles, where estimated grades are higher in lower grade areas and lower in higher grade areas. To quantify the level of smoothing of the estimates, further investigation is required (Section 14.7.5, Level of Smoothing/Variability).

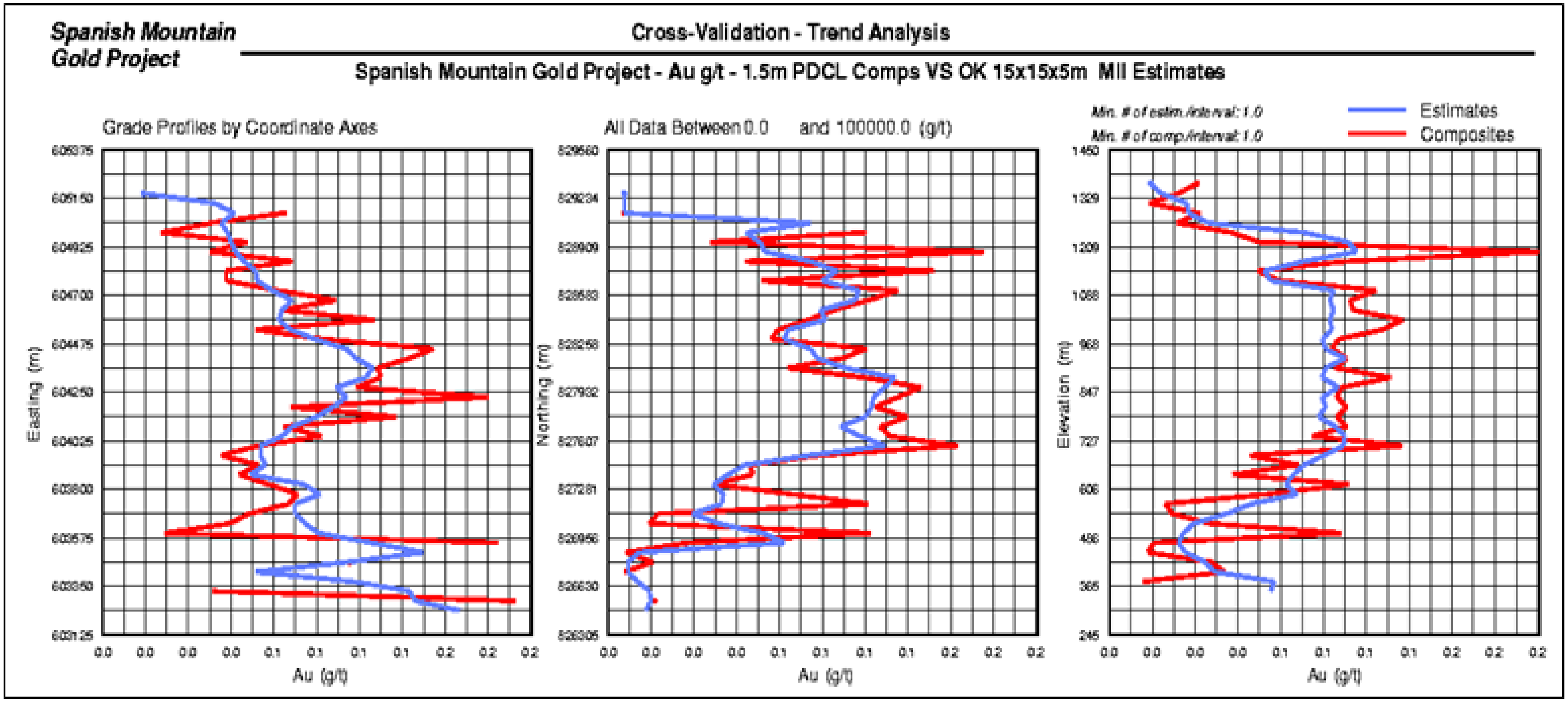


Figure 14-16 Gold Grade Profiles of Declustered Composites and Block Estimates – Measured, Indicated, and Inferred

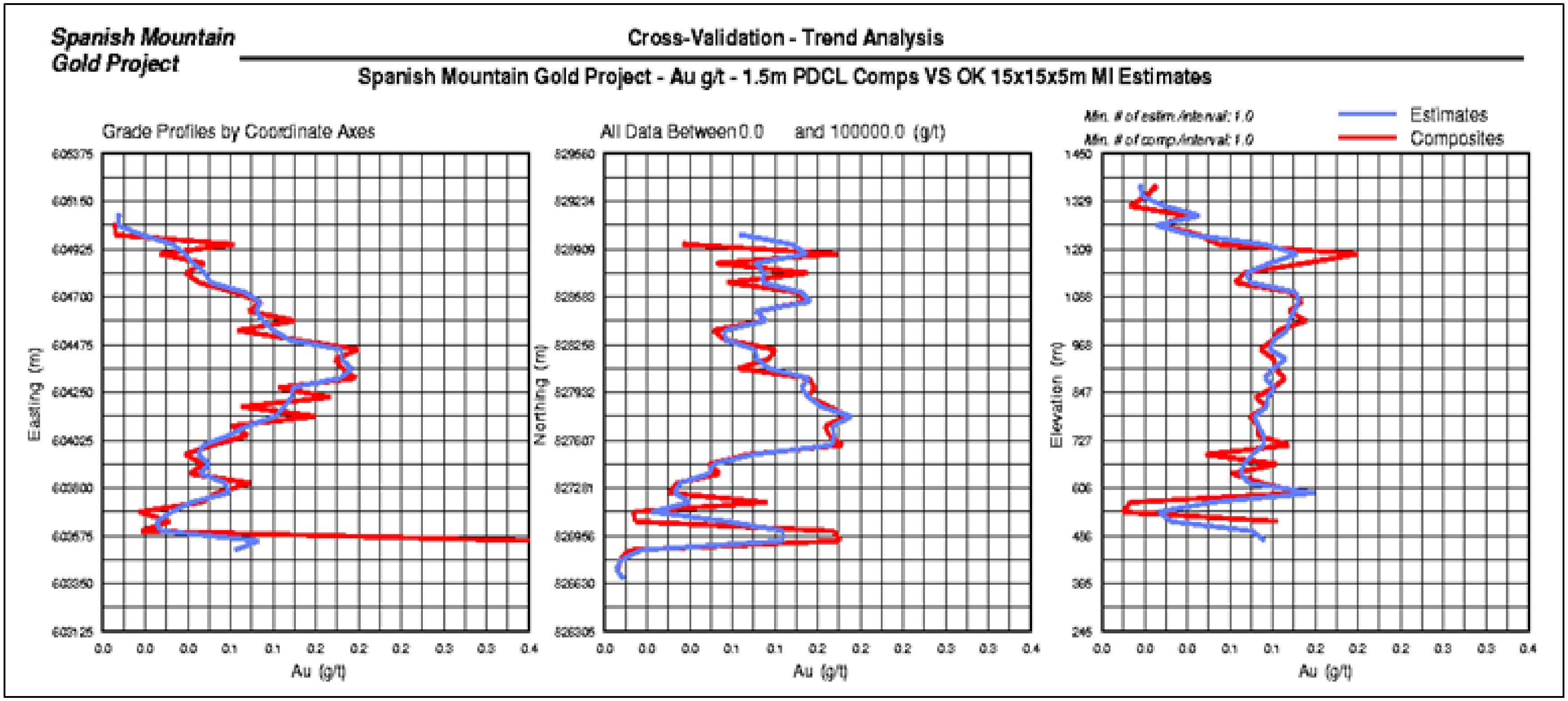


Figure 14-17 Gold Grade Profiles of Declustered Composites and Block Estimates – Measured and Indicated

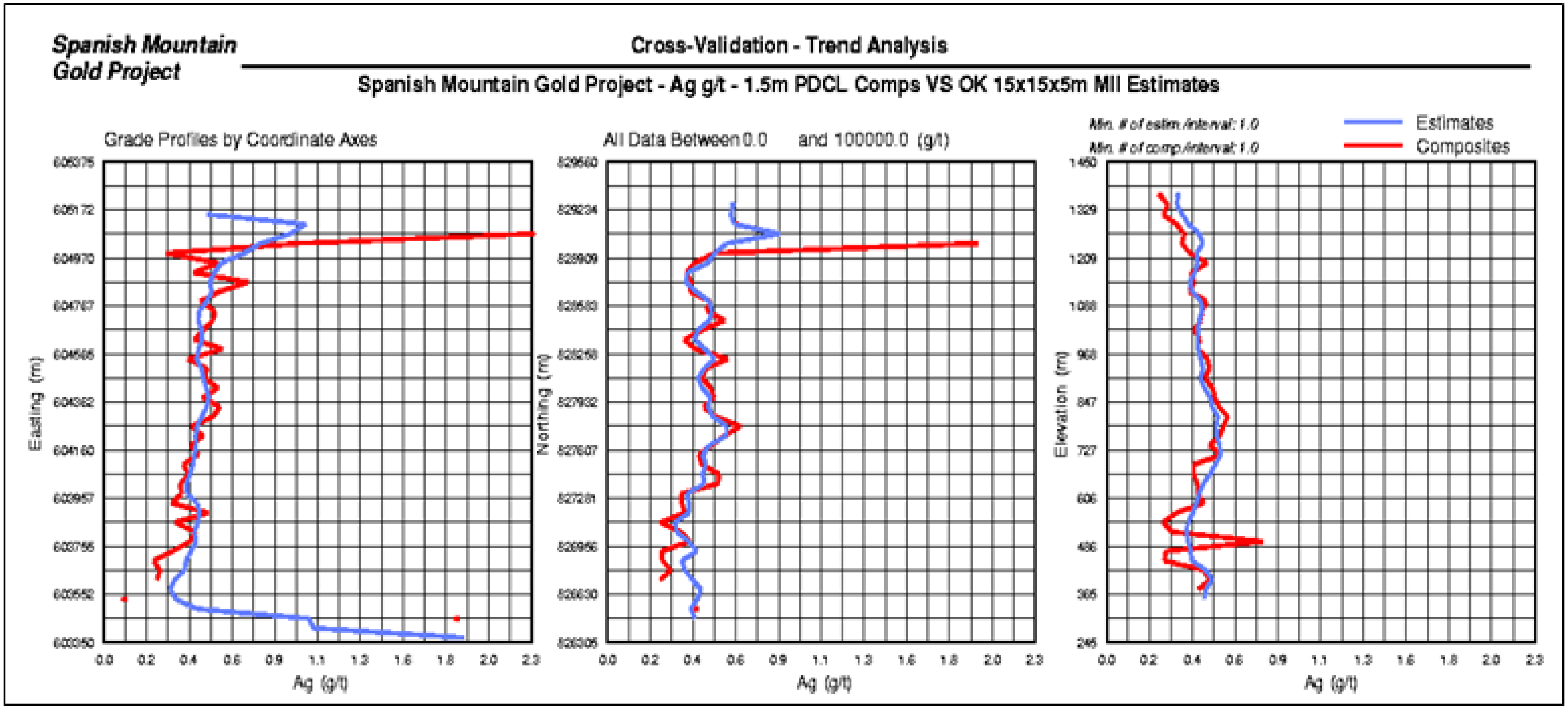


Figure 14-18 Silver Grade Profiles of Declustered Composites and Block Estimates – Measured, Indicated, and Inferred

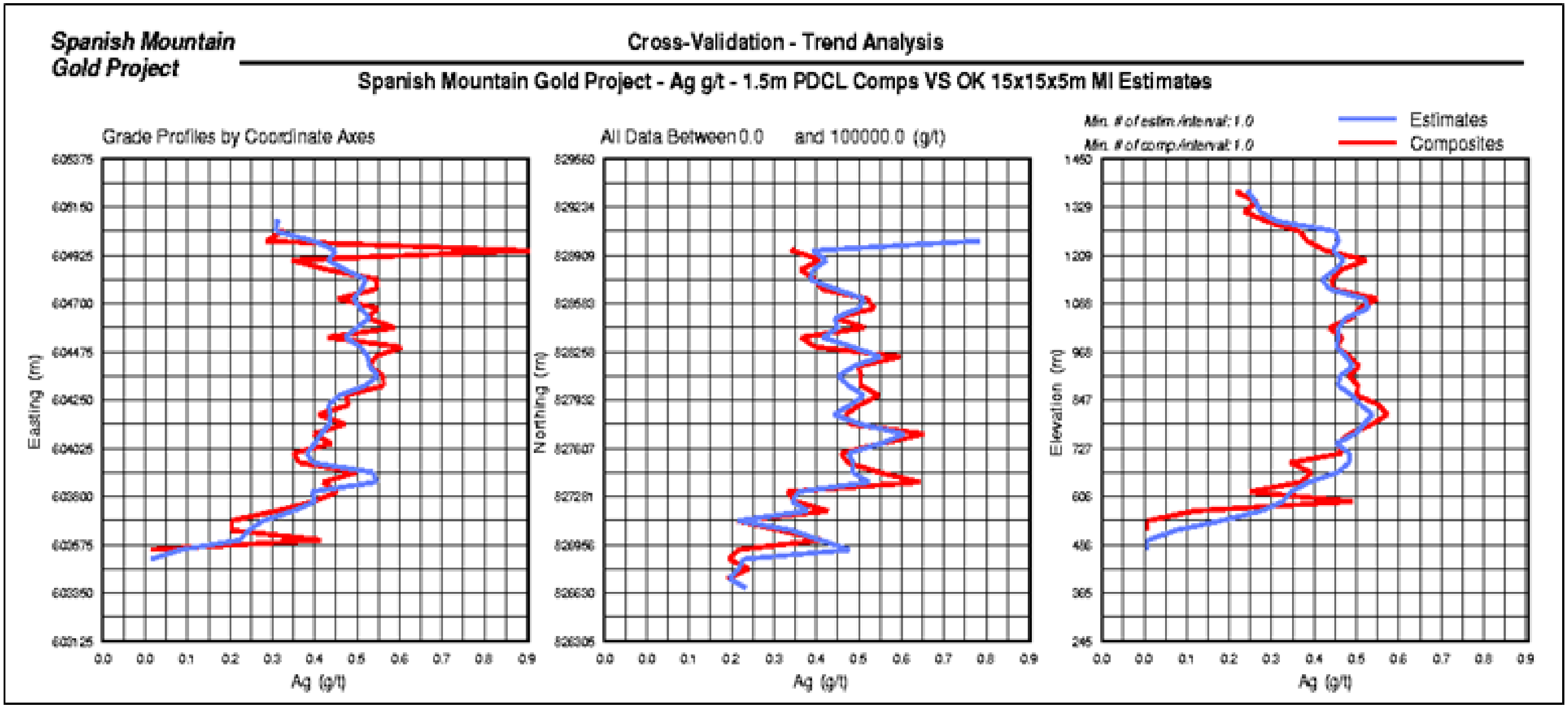


Figure 14-19 Silver Grade Profiles of Declustered Composites and Block Estimates – Measured and Indicated

14.7.5 Level of Smoothing/Variability

The level of smoothing/variability of the estimates can be measured by comparing a theoretical distribution of block grades with that of the actual estimates. The theoretical distribution of block grades is derived from that of the declustered composites, where a change of support algorithm is utilized for the transformation (Indirect Lognormal Correction). In this case, the variance of the composites' grade population is corrected (reduced) with the help of the variogram model, to reflect a distribution of block grades (15m x 15m x 5m). The comparison of the coefficient of variation (CV) of this population with that of the actual block estimates provides a measure of smoothing. Ideally a lower CV from the estimates by 5 to 30% is targeted as a proper amount of smoothing. This smoothing of the estimates is desired as it allows for the following factors: the imperfect selection of ore blocks at the mining stage (misclassification), the block grades relate to much larger volumes than the volume of core (support effect), and the block grades are not perfectly known (information effect). A CV lower than 5 to 30% for the estimates would indicate a larger amount of smoothing, while a higher CV would represent a larger amount of variability. Too much smoothing would be characterized by grade estimates around the average grade, where too much variability would be represented by estimates with abrupt changes between lower and higher-grade areas.

Results of the level of smoothing/variability analysis are presented in Table 14-15 for gold and silver. As observed in this Table, the CVs of the gold and silver estimates are within the targeted range, and for such represent an adequate amount of smoothing/variability.

Table 14-15 Level of Smoothing/Variability of Gold and Silver Estimates

Gold		
CV – Theoretical Block Grade Distribution	**CV – Actual Block Grade Distribution**	**Difference**
2.381	2.111	-11.3%
Silver		
CV – Theoretical Block Grade Distribution	**CV – Actual Block Grade Distribution**	**Difference**
0.932	0.869	-6.7%

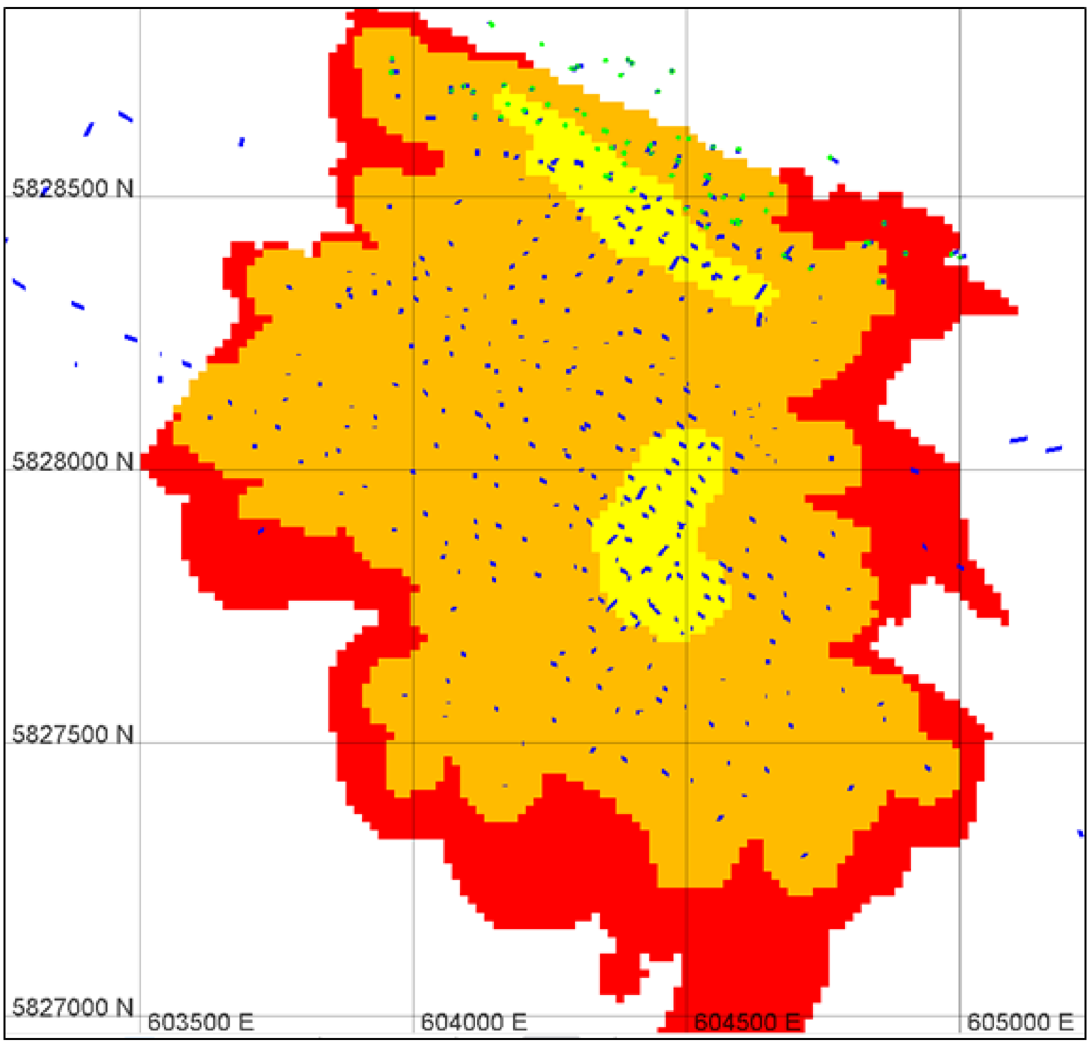
14.8 Resource Classification

The Mineral Resource was classified with a two-step approach. Firstly, the Mineral Resource was classified into Measured, Indicated, and Inferred classes based on the variogram ranges. The first range of the variogram identified Measured resources, the second range of the variogram identified Indicated resources, and Inferred resources were identified as beyond the second range of the variogram. The average distance of samples from the block center was utilized as the classification criterion. The classification distances for all mineralized domains are provided in Table 14-16.

Table 14-16 Classification Distances

Measured	Indicated	Inferred
≤ 15.0m	> 15.0m and ≤ 59.0m	> 59.0m

A second step in the classification of Mineral Resources consisted of adjusting the initial classification into more uniform areas for each class, based on the density of drilling. An example of the classification results is shown in Figure 14-20.



Source: Jutras, 2021

Figure 14-20 Mineral Resource Classification Categories: Measured (yellow), Indicated (orange), and Inferred (red) – Level 925El

14.9 Mineral Resource Calculation

The Mineral Resource was calculated for 15m (X) x 15m (Y) x 5m (Z) blocks with an average specific gravity value assigned to each mineralized domain. There is a total of 2,163 specific gravity measurements from which statistics were performed for each of the mineralized domains. The average specific gravity values assigned to all blocks within each domain are presented in Table 14-17.

Table 14-17 Average Specific Gravity by Domain

Domain	Specific Gravity
1-UARGS	2.762
2-UARGN	2.765
3-TUFFS	2.789
4-TUFFN	2.799
5-LARG	2.770
6-SLTSTS	2.774
7-SLTSTN	2.781
8-NZARG	2.784
9-OVB	2.100
ALL	2.777

For the mineral resource's tonnage, the block volume, and its proportion below the topography surface, along with its assigned specific gravity were utilized in the calculations.

With the objective to satisfy the NI 43-101 requirement of reporting a Mineral Resource that provides "reasonable prospects for economic extraction", a pit shell was optimized to constrain the Mineral Resources. A summary of the Mineral Resource pit-constraining parameters is shown in Table 14-18, and an example of the constraining pit shell is shown in Figure 14-21.

Table 14-18 Mineral Resource Pit Constraining Parameters

Parameters	
Gold Price	1,600.00 US$/oz
Silver Price	20.00 US$/oz
Mining Cost - Ore	3.20 $/t
Mining Cost - Waste	2.93 $/t
Processing Cost	7.33 $/t
G&A Cost	2.67 $/t
Exchange Rate	0.75 C$/$US
Gold Recovery	91%
Silver Recovery	25%
Pit Slopes	Variable angles from 21° to 35°

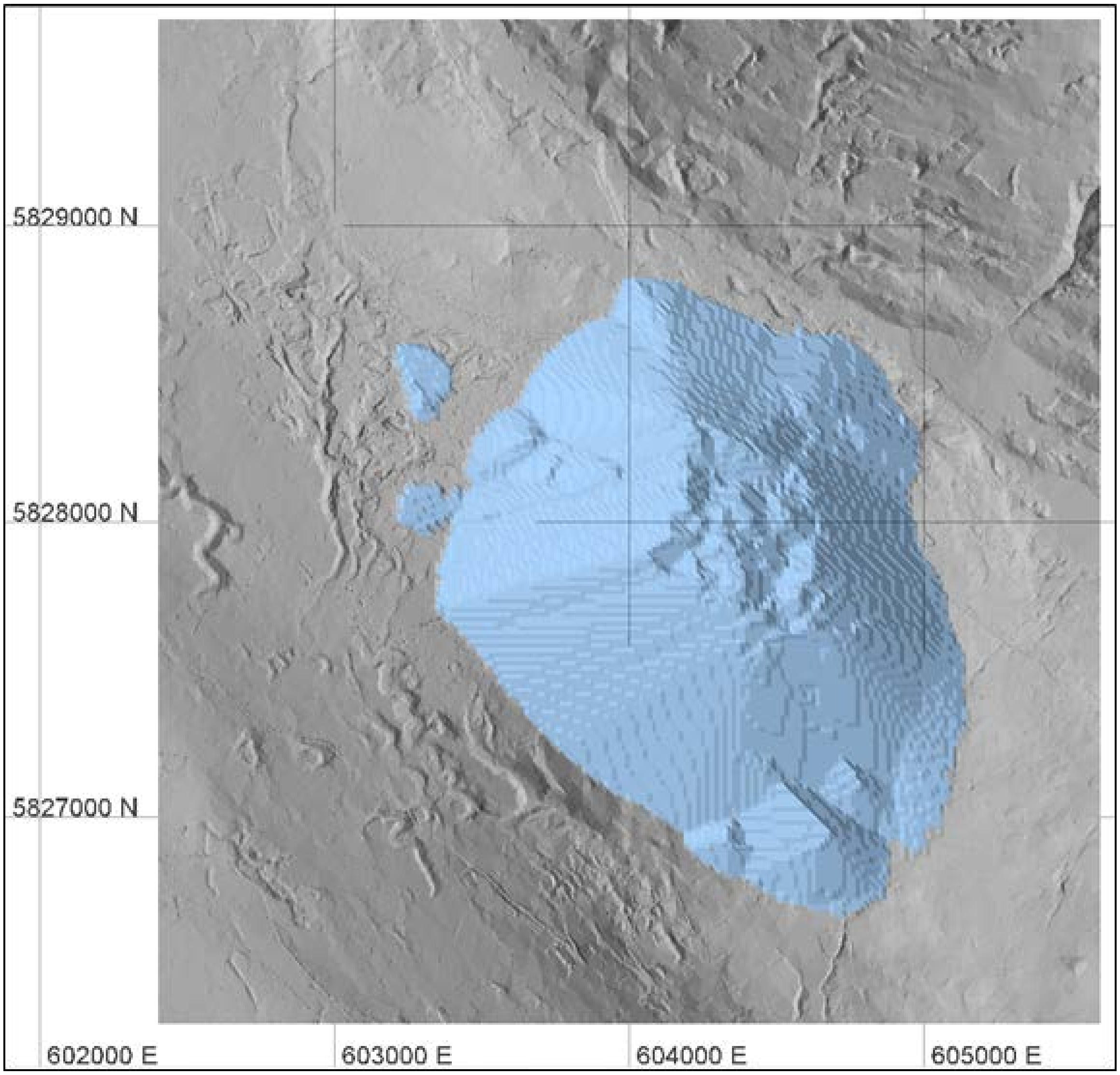


Figure 14-21 Mineral Resource Constraining Open Pit Shell – Plan View

The Measured, Indicated, and Inferred Mineral Resources are presented in Table 14-19 at a 0.15 g/t Au cut-off grade, and in Table 14-20 at various Au cut-off grades. The Mineral Resources are inclusive of Mineral Reserves in these statements. Grade-tonnage curves of the Mineral Resource are presented in Figure 14-22 for gold and in Figure 14-23 for silver.

It should be noted that Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources estimated will be converted into Mineral Reserves. The estimate of Mineral Resources may be materially affected by future changes in environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues. There are no currently known issues that negatively impact the stated Mineral Resources.

The CIM definitions were followed for the classification of Measured, Indicated, and Inferred Mineral Resources. The Inferred Mineral Resources have a lower level of confidence than that applying to Indicated Mineral Resources and must not be converted to mineral reserves. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

Table 14-19 Mineral Resources at a 0.15 g/t Au Cut-Off Grade

Classification	Tonnage Tonnes	Average Grade		Metal Content	
		Au g/t	Ag g/t	Au oz	Ag oz
Measured	68,429,000	0.59	0.67	1,289,000	1,474,000
Indicated	225,724,000	0.47	0.73	3,418,000	5,298,000
Measured+Indicated	294,153,000	0.50	0.72	4,707,000	6,772,000
Inferred	18,343,000	0.63	0.76	372,000	448,000

Notes:

1. *Mineral resources' tonnage and ounces have been rounded to the nearest thousand.*
2. *The Inferred mineral resources have a lower level of confidence than that applying to Indicated mineral resources and must not be converted to mineral reserves. It is reasonably expected that most Inferred mineral resources could be upgraded to Indicated mineral resources with continued exploration.*
3. *The mineral resources are constrained within a pit optimized at a $1,600US$/oz gold price and $20US$/oz silver price.*
4. *Inclusive of Mineral Reserves - Effective February 3, 2021.*

Table 14-20 Sensitivity of Mineral Resources to Various Au Cut-Off Grades

Classification	Au Cut-Off g/t	Tonnage tonnes	Average Grade		Metal Content	
			Au g/t	Ag g/t	Au oz	Ag oz
Measured	0.15	68,429,000	0.59	0.67	1,289,000	1,474,000
	0.20	57,829,000	0.66	0.68	1,229,000	1,264,000
	0.30	42,963,000	0.81	0.69	1,112,000	953,000
	0.40	32,923,000	0.95	0.70	1,000,000	741,000
	0.50	25,520,000	1.09	0.72	894,000	591,000
Indicated	0.15	225,724,000	0.47	0.73	3,418,000	5,298,000
	0.20	174,161,000	0.56	0.73	3,130,000	4,088,000
	0.30	111,594,000	0.74	0.72	2,637,000	2,583,000
	0.40	77,627,000	0.91	0.73	2,261,000	1,822,000
	0.50	58,389,000	1.06	0.75	1,984,000	1,408,000
Measured+Indicated	0.15	294,153,000	0.50	0.72	4,707,000	6,772,000
	0.20	231,990,000	0.58	0.72	4,359,000	5,352,000
	0.30	154,557,000	0.75	0.71	3,749,000	3,536,000
	0.40	110,550,000	0.92	0.72	3,261,000	2,563,000
	0.50	83,909,000	1.07	0.74	2,878,000	1,999,000
Inferred	0.15	18,343,000	0.63	0.76	372,000	448,000
	0.20	14,899,000	0.74	0.79	353,000	378,000
	0.30	6,824,000	1.30	0.77	285,000	169,000
	0.40	3,874,000	2.03	0.87	253,000	108,000
	0.50	3,111,000	2.42	0.92	242,000	92,000

**Mineral Resources' tonnage and ounces have been rounded to the nearest thousand.*
**Effective February 3, 2021*

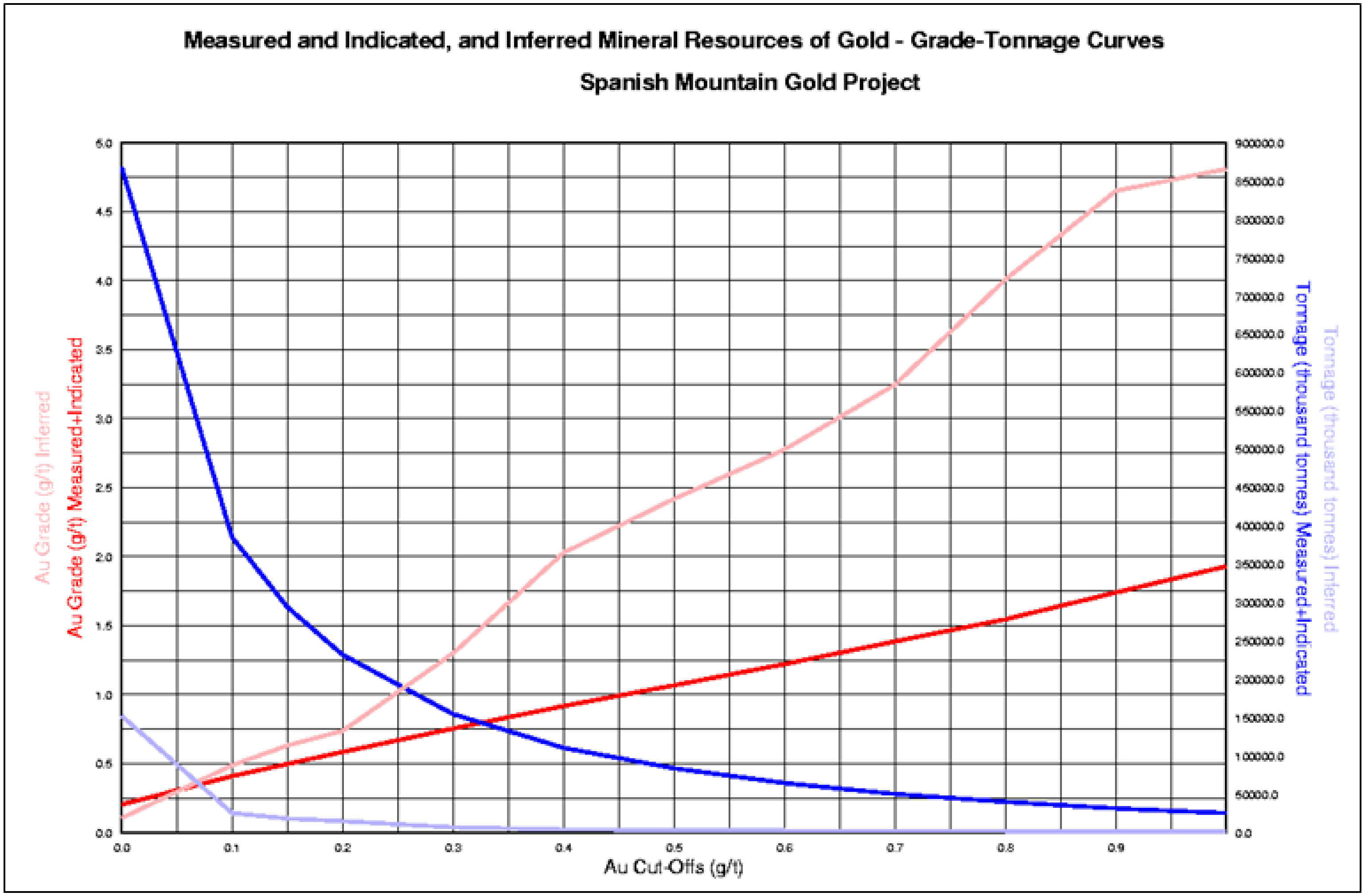


Figure 14-22 Gold Grade-Tonnage Curves of the Measured and Indicated, and Inferred Mineral Resources

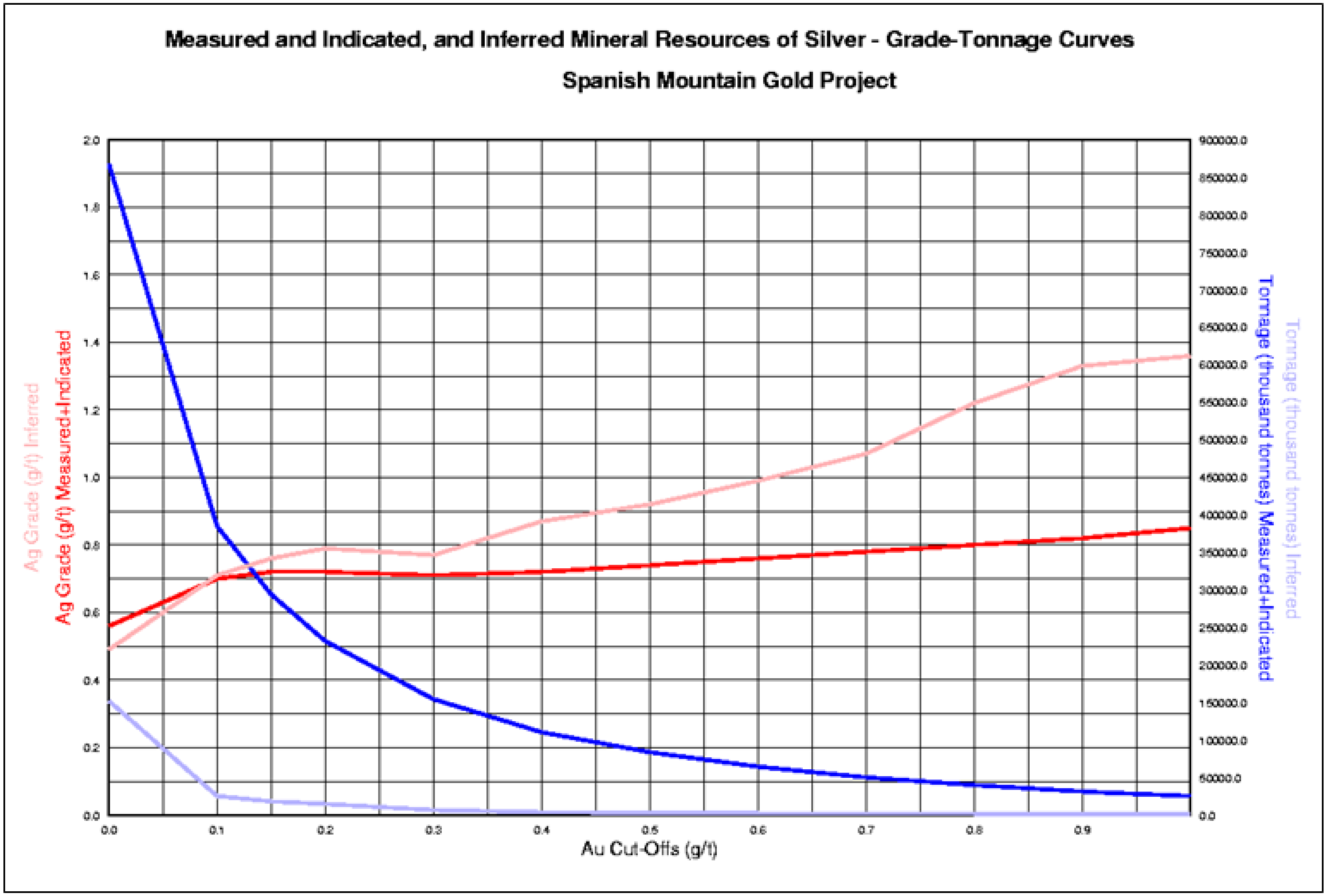


Figure 14-23 Silver Grade-Tonnage Curves of the Measured and Indicated, and Inferred Mineral Resources

14.10 Comparison with the 2019 PEA Mineral Resources

The current estimation of the Mineral Resources of the Spanish Mountain Gold Project is compared to the 2019 estimation of the Mineral Resources from the Preliminary Economic Analysis (PEA) in Table 14-21.

Table 14-21 Comparison of Mineral Resources* from the 2019 PEA and 2021 PFS at a 1.5 g/t Au Cut-Off Grade

Classification	Study	Tonnage	Average Grade		Metal Content	
		tonnes	Au g/t	Ag g/t	Au oz	Ag oz
Measured	2021 PFS	68,429,000	0.59	0.67	1,289,000	1,474,000
	2019 PEA	29,600,000	0.60	0.83	569,000	791,000
	Difference	131.2%	-1.7%	-19.3%	126.5%	86.3%
Indicated	2021 PFS	225,724,000	0.47	0.73	3,418,000	5,298,000
	2019 PEA	243,600,000	0.46	0.69	3,566,000	5,413,000
	Difference	-7.3%	2.2%	5.8%	-4.2%	-2.1%
Measured+Indicated	2021 PFS	294,153,000	0.50	0.72	4,707,000	6,772,000
	2019 PEA	273,200,000	0.47	0.71	4,135,000	6,204,000
	Difference	7.7%	6.4%	1.4%	13.8%	9.2%
Inferred	2021 PFS	18,343,000	0.63	0.76	372,000	448,000
	2019 PEA	52,400,000	0.37	0.67	619,000	1,128,000
	Difference	-65.0%	70.3%	13.4%	-39.9%	-60.3%

*Mineral Resources' tonnage and ounces have been rounded to the nearest thousand.

From Table 14-21, an increase of the Measured and Indicated, and decrease of the Inferred Mineral Resources are observed for the current estimate. These differences stem from changes in the grade estimation strategy (OK in 2021 vs MIK+ID^3 in 2019), changes in the Mineral Resource classification criteria, and changes in the pit constraining parameters.

14.11 Grade Estimation of As, Ca, and Total S

An estimation of the arsenic (As), calcium (Ca), and total sulphur (S) grades was carried out subsequently to the estimation of gold and silver grades. The main objective of these estimates was to provide models of grade distributions for environmental and metallurgical purposes. A total of 124,695 As assays in ppm from 970 holes, 125,552 Ca assays in % from 981 holes, and 56,848 total S assays in % from 454 holes were available from the original drillhole database. Assays were then composited to 1.5m lengths within the mineralized domains. The capping of higher-grade outliers was examined, and composites were capped for each mineralized domain to thresholds varying from 280 ppm to 700pm for As, from 8% to 10% for Ca, and from 4% to 8% for total S. A variographic analysis was performed for all composites within mineralized domains for each element of interest. Directions of better grade continuity were established to be along strike and down dip, with strike orientations of 130° for As and Ca, and 125° for total S, and down dip orientations of 80° northeast for As, vertical for Ca, and 60° northeast for total S. The ordinary kriging estimation technique on capped 1.5m composites served as the grade interpolation approach for As, Ca, total S. A set of two estimation runs with a minimum of two and maximum of twelve samples was utilized. Similarly, to the Au and Ag estimates, various validation tests were performed on the As, Ca, and total S estimates with results showing adequate grade estimates.

14.12 Discussion and Recommendations

The estimation of the Spanish Mountain Gold Project's Mineral Resources for the Prefeasibility Study was carried out with the same drillhole data and geologic model as for the 2019 PEA study, as no new assays were available and no new geologic information was acquired since then. All other aspects of the Mineral Resource estimation process were re-visited for the Prefeasibility Study. An ordinary kriging grade interpolation technique with capped composites was selected as the grade estimation approach for gold and silver. A multiple indicator kriging method for gold and inverse distance cubed method for silver were previously used for the 2019 PEA study.

Approximately 20% of the drillholes utilized for this study are from reverse circulation drilling (RC), with the remainder of holes being diamond drillholes (DDH). Comparative statistics between the RC and DDH gold assays have shown higher average gold grades from the RC holes. Further investigation did not indicate downhole contamination from the RC drilling. Marc Jutras is satisfied that the much larger sample derived from the RC drilling in an environment of coarse gold in friable material provides a better grade approximation.

The higher gold grade variability denoted by the high coefficients of variation from the 1.5m composites could be indicative of a grade estimation technique where higher grades are treated differently than lower grades, such as a multiple indicator kriging method. However, from the capping analysis the high coefficients of variation were reduced to acceptable levels for an ordinary kriging method to be utilized.

An independent validation of the drillhole database was carried out by Marc Jutras prior to the estimation of the Mineral Resources. In this exercise approximately 10% of the gold assays were checked against the assay certificates. Similarly, approximately 10% of the drillhole collar coordinates and downhole surveys were checked against the drill logs. Overall, no significant errors were found and the drillhole database was deemed valid for the estimation of Mineral Resources.

During the drillhole database validation exercise, a survey of drillhole collars carried out in 2008 by Allnorth Consultants Limited indicated different collar coordinates for 48 holes. From further investigation it was found that these holes were re-surveyed after this initial survey and that the collar coordinates in the drillhole database are the correct ones. Survey checks for a few drillholes carried in December 2020 have confirmed the current coordinates. It is recommended that all the 48-hole collars be re-surveyed during the 2021 summer months and documented to further confirm their collar coordinates.

Marc Jutras is satisfied that the current geology model is adequate for the estimation of Mineral Resources, however it is suggested that the broader geologic units could be refined for future updates.

A drilling campaign was carried out in November and December of 2020 and was not incorporated into this Mineral Resource estimate due to delays in the assaying process. It is recommended that an update of the Mineral Resource estimate be carried out with these additional drillholes once the results have been validated.

The abundance of drillhole data at a relatively close spacing allowed for conclusive variogram models for gold and silver. This brings greater certainty to the assessment of the gold and silver grade continuity.

The satisfactory results obtained from the various validation tests for the gold and silver estimates indicate an adequate representation of the Spanish Mountains Gold Project's Mineral Resource, based on the current geologic understanding and available information.

15 MINERAL RESERVE ESTIMATES

15.1 Introduction

The Mineral Reserves for the Spanish Mountain Gold Project are a subset of the Measured and Indicated Mineral Resources, described in Item 14, and supported by Prefeasibility Study engineering described in subsequent sections of this Technical Report, including the mine engineering summarized in Item 16.

15.2 Mineral Reserves Statement

Proven and Probable Mineral Reserves have been modified from Measured and Indicated Mineral Resources at Spanish Mountain Gold and are summarised in Table 15-1. Inferred Mineral Resources are set to waste. Mineral Reserves are supported by 2021 Prefeasibility Study engineering and include modifying factors appropriate for the planned mining methods, including mining loss and dilution. Mineral Reserves have been estimated using the CIM 2019 Best Practices Guidelines (CIM, 2019) and are classified using the 2014 CIM Definition Standards (CIM, 2014).

Table 15-1 Proven and Probable Mineral Reserves

Reserve Class	Mill Feed (Mt)	Mill Feed Gold Grade (g/t)	Contained Metal (Moz)	Mill Feed Silver Grade (g/t)	Contained Metal (Moz)
Proven	40.8	0.79	1.03	0.67	0.88
Probable	55.1	0.74	1.31	0.74	1.31
Total	**95.9**	**0.76**	**2.34**	**0.71**	**2.19**

Notes:

1. *The Mineral Reserve estimates were prepared by Marc Schulte, P.Eng. (who is also an independent Qualified Person), reported using the 2014 CIM Definition Standards, and have an effective date of March 31, 2021.*
2. *Mineral Reserves are mined tonnes and grade; the reference point is the mill feed at the primary crusher.*
3. *Mineral Reserves are reported at a cut-off grade of 0.30 g/t Au. Cutoff grade assumes US$1,500/oz. Au at a currency exchange rate of 0.76 US$ per C$; 99.8% payable gold; US$5.00/oz. offsite costs (refining and transport); and uses an 80% low grade metallurgical recovery. The cut off-grade covers processing costs of $6.50/t, site and administrative (G&A) costs of $2.50/t, coverage for project sustaining capital costs of $3.00/t, and a stockpile rehandle cost of $2.00/t.*
4. *Mined tonnes and grade are based on an SMU of 15 m x 15 m x 5 m, including additional estimates for mining loss (3%) and dilution between ore and waste zones (6.6%, 0.24 g/t Au, 0.60 g/t Ag).*
5. *Numbers have been rounded as required by reporting guidelines.*

Mineral Reserves within Pit Phases

Open pits are based on the results of Pseudoflow sensitivity analysis, and then designed into detailed pit phases to develop pit reserves for mine production scheduling. The Mineral Reserves by designed pit phase are shown in Table 15-2.

15.3 Factors that May Affect the Mineral Reserve Estimate

Mineral Reserves are based on the engineering and economic analysis described in Chapters 16 to 22 of this report. Changes in the following factors and assumptions may affect the Mineral Reserve estimate:

- metal prices,
- interpretations of mineralisation geometry and continuity of mineralisation zones,
- geotechnical and hydrogeological assumptions,
- ability of the mining operation to meet the targeted annual production rate,
- operating cost assumptions,
- mining and process plant recoveries,
- ability to meet and maintain permitting and environmental license conditions, and the ability to maintain the social license to operate.

Table 15-2 Proven and Probable Mineral Reserves within Designed Pit Phases

Pit Phase	P621	P622i	P623i	P624i	P625i	P626i	P627i	P628i	P629i	Total
Measured Resource (kt)	14,052	9,794	0	603	3,298	11	0	4,148	8,908	40,814
Gold Grade (g/t)	0.87	0.82	0.00	0.69	0.62	0.43	0.00	0.77	0.70	0.79
Indicated Resource (kt)	12,258	10,373	1,031	2,123	3,526	2,130	986	6,154	16,532	55,113
Gold Grade (g/t)	0.67	0.68	3.11	0.60	0.52	0.77	0.60	0.65	0.78	0.74
Wasted Inferred Resource (kt)	93	67	1,036	4	284	506	85	4	558	2,637
Gold Grade (g/t)	0.53	0.53	3.13	0.55	0.52	0.70	0.40	0.36	0.50	1.57
Waste (kt)	50,524	56,010	40,185	3,378	20,855	20,751	3,256	33,216	151,782	207,792
Mill Feed (kt)	26,310	20,167	1,031	2,726	6,824	2,141	986	10,302	25,440	95,927
Gold Grade (g/t)	0.78	0.74	3.11	0.62	0.57	0.77	0.60	0.70	0.75	0.76
Silver Grade (g/t)	0.70	0.69	1.27	0.95	0.83	0.71	0.49	0.68	0.67	0.71
Waste and Inferred (kt)	50,930	56,077	41,221	3,382	21,139	21,257	3,341	33,220	152,340	382,907
Strip Ratio	1.94	2.78	39.98	1.24	3.10	9.93	3.39	3.22	5.99	3.99
Total Pit Contents (kt)	77,240	76,244	42,252	6,108	27,963	23,398	4,327	43,522	177,780	478,834

Notes:

1. *A cut-off grade of 0.30 g/t Au is applied.*
2. *Mined tonnes and grade are based on an SMU of 15 m x 15 m x 5 m, including additional estimates for mining loss (3%) and dilution between ore and waste zones (6.6%, 0.24 g/t Au, 0.60 g/t Ag).*
3. *Mineral Reserves in this table are not additive to the Mineral Reserves in Table 15-1. Footnotes to Table 15-1 apply to this table.*

16 MINING METHOD

The Mineral Reserves stated in Item 15 are supported by the open pit mine plan summarised in this chapter.

Open pit mine designs, mine production schedules and mine capital and operating costs have been developed for the Spanish Mountain Gold deposit at a Prefeasibility level of engineering.

16.1 Summary

The open pit is designed for approximately 14 years of operations. The ROM production contained within the designed open pit, summarized in Table 16-1 with a 0.30 g/t gold cut-off, forms the basis of the mine plan and production schedule.

Table 16-1 Mine plan ROM Production

	Unit	Amount
Mill Feed	kt	95,927
Gold Grade	g/t	0.76
Silver Grade	g/t	0.71
Waste Material	kt	382,907
Strip Ratio	t/t	4.0

The crusher will be fed with material from the pit and supplemented by the low-grade stockpile in years 13 and 14, at an average feed rate of 20,000 t/d.

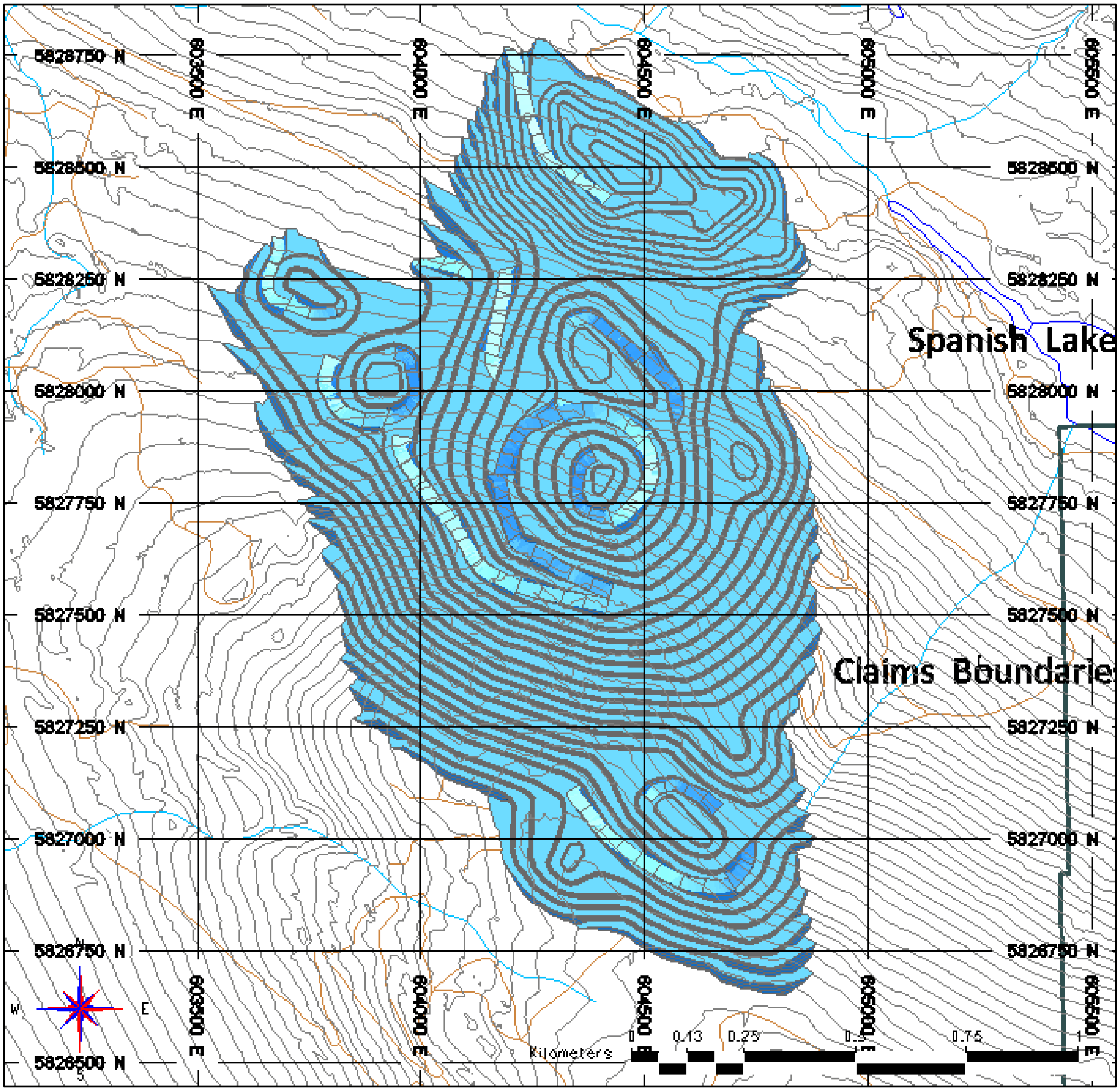
Figure 16-1 shows a plan view of the design for the ultimate pit.

To develop the most economic feed to the mill in the early years, and to provide a smooth transitional stripping plan for the duration of the LOM, open pit mining is scheduled from nine mining phases. Phase 1 will commence near the centre of the deposit, where the highest grade of mineralized resource and lowest strip ratio will be encountered.

To enhance the economics of the project an elevated 0.40 g/t cut-off grade is employed in the initial years. Mineralized material that is below the elevated cut-off grade, but able to cover the cost of milling and handling once it is hauled out of the pit, will be sent to a stockpile near the crusher and reclaimed at the end of the mine life or blended with the run-of-mine (ROM) feed if an appropriate opportunity arises.

Most the pit waste material will be hauled to stockpiles located on the west side of the pit. Preliminary geochemistry studies on the pit rock indicate that most of the pit waste rock is non-potentially acid generating (NPAG). The remaining potentially acid generating (PAG) waste rock will be sub-aqueously disposed in the tailings pond. Suitable pit waste rock will also be hauled to the TMF for dam construction, as needed.

The General Arrangement, Figure 1-4 shows the mine layout for the pit, stockpiles, and haul roads.



Source: Moose Mountain, 2021

Figure 16-1 Ultimate Pit Design - Plan View

16.2 Key Design Criteria

16.2.1 Mine Planning 3D Block Model

Mine planning work is based on the resource 3DBM as described in Item 14. Extra items are added to the resource 3DBM to carry out open pit mine planning.

16.2.2 Mining Loss and Dilution

The mineralized material is represented in the block model on a whole block basis. The 15 m x 15 m x 5 m size blocks are large mining units; averaging of the metal grade over an entire block suggests that the grade may be considerably smoothed resulting in significant internal model dilution.

For the purposes of this study, MMTS assumed that the selected mining fleet will effectively extract the mineralized material from the waste rock, and that mining dilution under normal situations will be offset by the modelling dilution.

Additional dilution is applied to the modelled gold and silver grades based on the number of waste/ore contact edges that are identified in each block at a 0.30 g/t gold cut-off grade. For each edge, a diluting wedge is assumed that carries a gold grade of 0.24 g/t and 0.60 g/t silver (based on measurements of grade surrounding these edge blocks). Table 16-2 below lists the dilution %'s estimated based on the number of contact edges. Applying these percentages to the contact blocks has an effect of diluting out the overall in-pit resource by and additional 6.6%.

Table 16-2 Dilution % based on number of waste/ore contacts in block

Number of waste/ore contact edges (using 0.30 g/t gold cut-off)	Dilution % applied to block
1	4%
2	9%
3	14%
4	19%

A 97% mining recovery (3% loss) is also applied to account for operating challenges and inefficiencies such as excessive blast heave, carry-back in truck boxes due to wet material, misdirected materials, and other unforeseen exceptions.

A thorough modelling evaluation and geostatistical analysis is necessary to better understand and quantify the internal dilution. This analysis will be undertaken at the next level of study.

16.2.3 Datum

Topography is based on a LiDAR survey of the region including the entire extents of the Mineral Resource and subset Mineral Reserves.

16.2.4 Resource Class

Only Measured and Indicated Resource class materials are included as economic in the open pit mine plan. Inferred Resource class material is treated as waste rock.

16.2.5 Net Smelter Price and Net Smelter Return

Net Smelter Price (NSP) is used in place of the Market Price for gold and silver, to consider all offsite costs to the project and the combined value of both metals.

Using market prices of US$1,500/oz gold and US$20/oz silver results in NSP values of C$1,935/oz or C$62.22/g gold and C$0.69/g silver. The NSP calculation uses the inputs shown in Table 16-3:

Table 16-3 NSP Calculation Inputs

Description	Values	Units
Gold Price	$1,500.00	US$/oz
Silver Price	$20.00	US$/oz
Exchange rate	0.76	US$/C$
Payable Au	99.8%	%
Payable Ag	9.0%	%
Au Offsites (Refining/Transport)	$4.00	US$/oz
Ag Offsites (Refining/Transport)	$2.00	US$/oz
Royalty	1.5%	

Net Smelter Return (NSR) is defined as dollar value of metal at the mine gate and is treated as a grade value in the mine planning block model. NSR is the product of NSP, the metal grade in the block, and the process recovery.

16.2.6 Process Recovery

The process recovery assumptions for pit optimization runs are 91% for gold and 25% for silver. Lower process recovery assumptions of 80% gold and 0% silver are used for cut-off grade calculations.

16.2.7 Cut-off Gold Grade

The cut-off grade is chosen as the gold grade required to pay for processing costs, general site and administration costs, stockpile reclaim costs, and coverage for the project’s sustaining capital cost spend. The sum of these costs is estimated to be $15.00/t. Based on the NSP and process recovery formulas above, the economic gold cut-off grade is 0.30 g/t.

To boost mill feed grades, an elevated gold cut-off grade of 0.40 g/t is also applied early in the mine life. Material between the economic cut-off grade and this elevated cut-off grade is stockpiled and reclaimed back to the mill for processing at the end of the mine life.

16.2.8 Pit Slope Angles

The pit slopes design inputs are based on analysis and recommendations developed from geotechnical drilling and core logging carried out from 2010 to 2012 (BGC, 2021).

Design criteria are based on a geological model of the rock- type grouping and three main faults. Structural domains and discontinuity sets have been interpreted based on the current fault model, oriented core and televiewer discontinuity data, outcrop mapping data, and regional structural geology interpretations.

The pit wall angles are limited generally by the orientations of the structural discontinuities in the rock mass and vary significantly depending on the design sector. The bench and inter-ramp designs are based on kinematic analysis of the design discontinuity sets, assessment of bench crest back-break and resulting debris retention requirements, and 2D limit equilibrium analysis of inter-ramp slope stacks with consideration of the geological model, rock mass properties, cross-cutting anisotropic weakness, and fault zones.

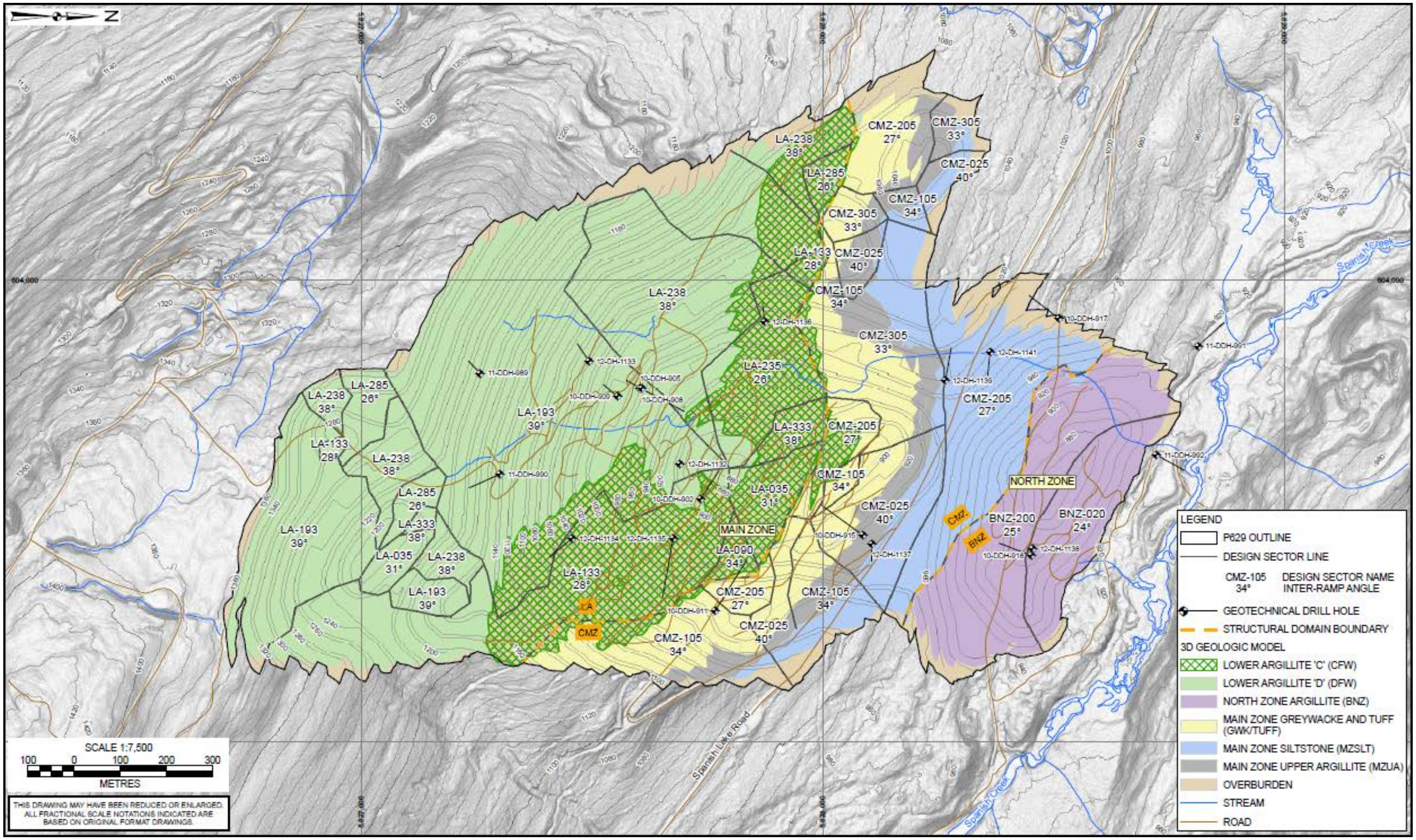
The majority of the recommended inter-ramp angles are controlled by discontinuity sets associated with faulting or bedding. Discontinuity sets applied to the inter-ramp scale analyses include only sets that appear in multiple data sources, are associated with mine-scale folding or faulting, or are observed to have shear-strength reducing characteristics like slickensides or infill.

Table 16-4 and Figure 16-2 summarize the relevant information, specifically the wall design criteria for each pit sector. In-pit ramps (29 m wide) and geotechnical berms (minimum 20 m wide) are included in the design where necessary to reduce the overall slope angles and facilitate geotechnical instrumentation and dewatering.

The Spanish Mountain Gold PFS pit optimization shells conform to shell overall slopes provided in Table 16-4. The detailed pit designs conform to all other listed criteria.

Table 16-4 Pit Slope Design Recommendations

Domain	Design Sector	Azimuth Start (°)	Azimuth End (°)	Shell Overall Slopes (°)	Bench Height (m)	Bench Face Angle (°)	Berm Width (m)	Geotechnical Berm Maximum Spacing (m)	Inter Berm Angle (°)
BNZ	BNZ-200	60	340	23	10	65	16.8	100	25
BNZ	BNZ-020	340	60	22	10	65	17.8	100	24
LA	LA-193	175	220	35	20	65	15.4	100	39
LA	LA-238	220	255	34	20	65	16.3	100	38
LA	LA-285	255	325	24	20	65	31.7	100	26
LA	LA-333	325	350	34	20	65	16.3	100	38
LA	LA-035	350	80	29	20	65	24.0	100	31
LA	LA-090	80	100	31	20	65	20.3	100	34
LA	LA-133	100	175	26	20	65	28.3	160	28
CMZ	CMZ-205	140	270	25	20	65	29.9	160	27
CMZ	CMZ-305	270	340	30	20	65	21.5	160	33
CMZ	CMZ-025	340	70	27	20	65	17.6	160	29
CMZ	CMZ-105	70	140	31	20	65	20.4	160	34
Overburden		0	360	21	10	65	-	-	21



Source: BGC, 2021

Figure 16-2 Structural Domains for Pit Slope Designs

16.3 Open Pit Optimization

Economic pit limits for this study of the Spanish Mountain deposit are determined using a Pseudoflow open pit evaluation.

The economic pit limit is selected after evaluating various pit shell cases. Each case represents the pit shell resulting from a different set of economic assumptions and pit slope inputs. The pit limit is chosen where incrementally larger pits produce marginal or negative economic returns.

16.3.1 Pit Optimization Operating Costs

Potential block revenues are calculated based on the NSP, process recovery, gold grade and densities within each block.

Operating costs are used in conjunction with these potential block revenues to run the pit optimization algorithm and generate open pit shells (Table 16-5). The following operating costs are used in the analysis:

Table 16-5 Pit Optimization Operating Cost Inputs

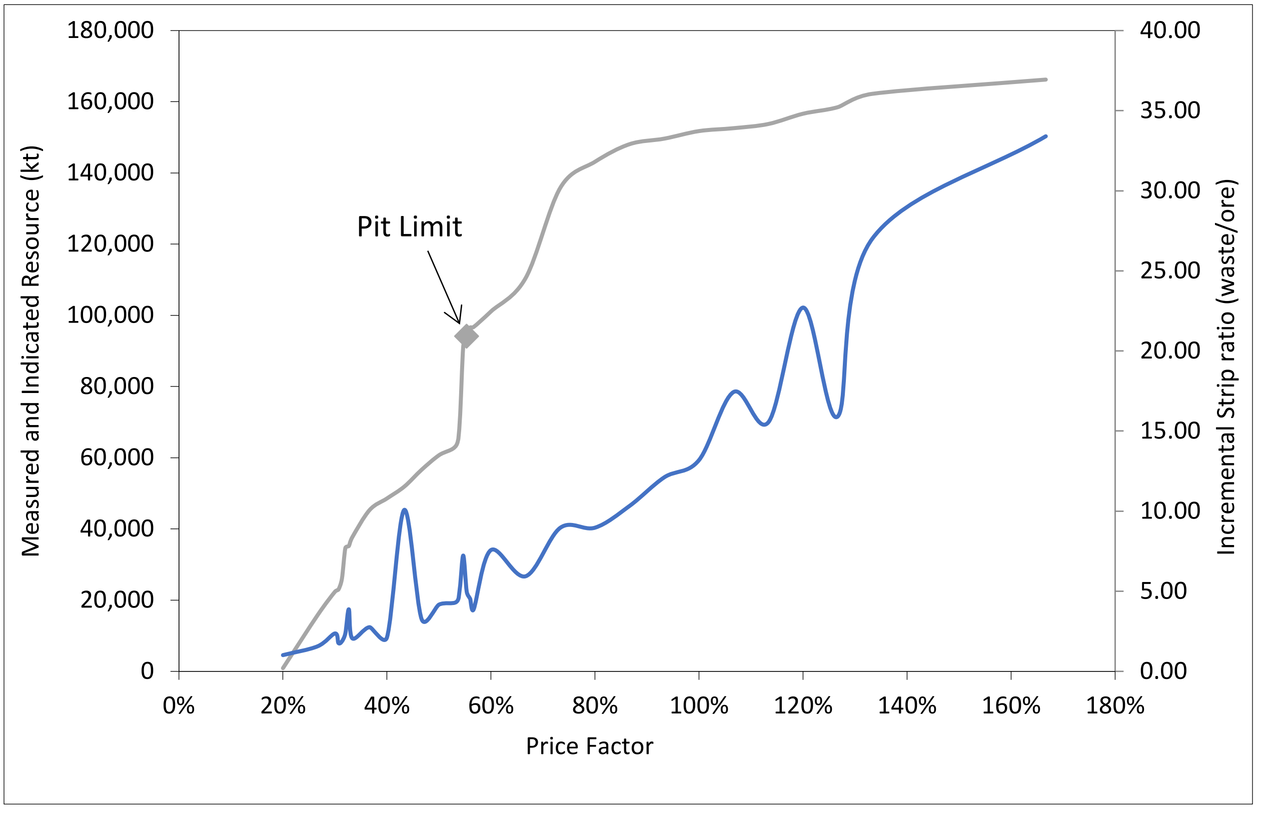
Operation	Cost
Base Ore Mining Cost (Pit Rim)	\$2.40/t
Base Waste Mining Cost (Pit Rim)	\$2.20/t
Incremental Haulage Cost	\$0.015/5 m bench below 950 m model elev.
Processing Cost	\$5.00/t
General/Administration Cost	\$2.00/t

Note that the pit optimisation analysis was re-run with final study costs and process recoveries, which yielded identical selected pit shells as described below.

16.3.2 Price Case Results

The economic pit limits are derived from the cost and price assumptions described above. By varying the input gold prices from US\$300/oz to US\$2,500/oz, while keeping metallurgical recoveries, operating costs and pit slopes constant at the values shown above, various generated pit cases are evaluated to determine the point at which incremental pit shells produce marginal or negative economic returns. This drop-off is due to increasing strip ratios, decreasing gold grades and increased mining costs associated with the larger pit shells. Note: this is not a price sensitivity of the economic pit limit since the cut-off grade is not varied for each pit shell.

Figure 16-3 shows the potential inventory contents of the generated Price Case pit shells using a 0.30 g/t cut-off gold grade. Various inflection points can be seen in the curve drawn of cumulative inventory by pit case.



Source: Moose Mountain, 2021

Figure 16-3 Price Cases Cumulative Inventory

16.3.3 Selected Ultimate Pit Limits

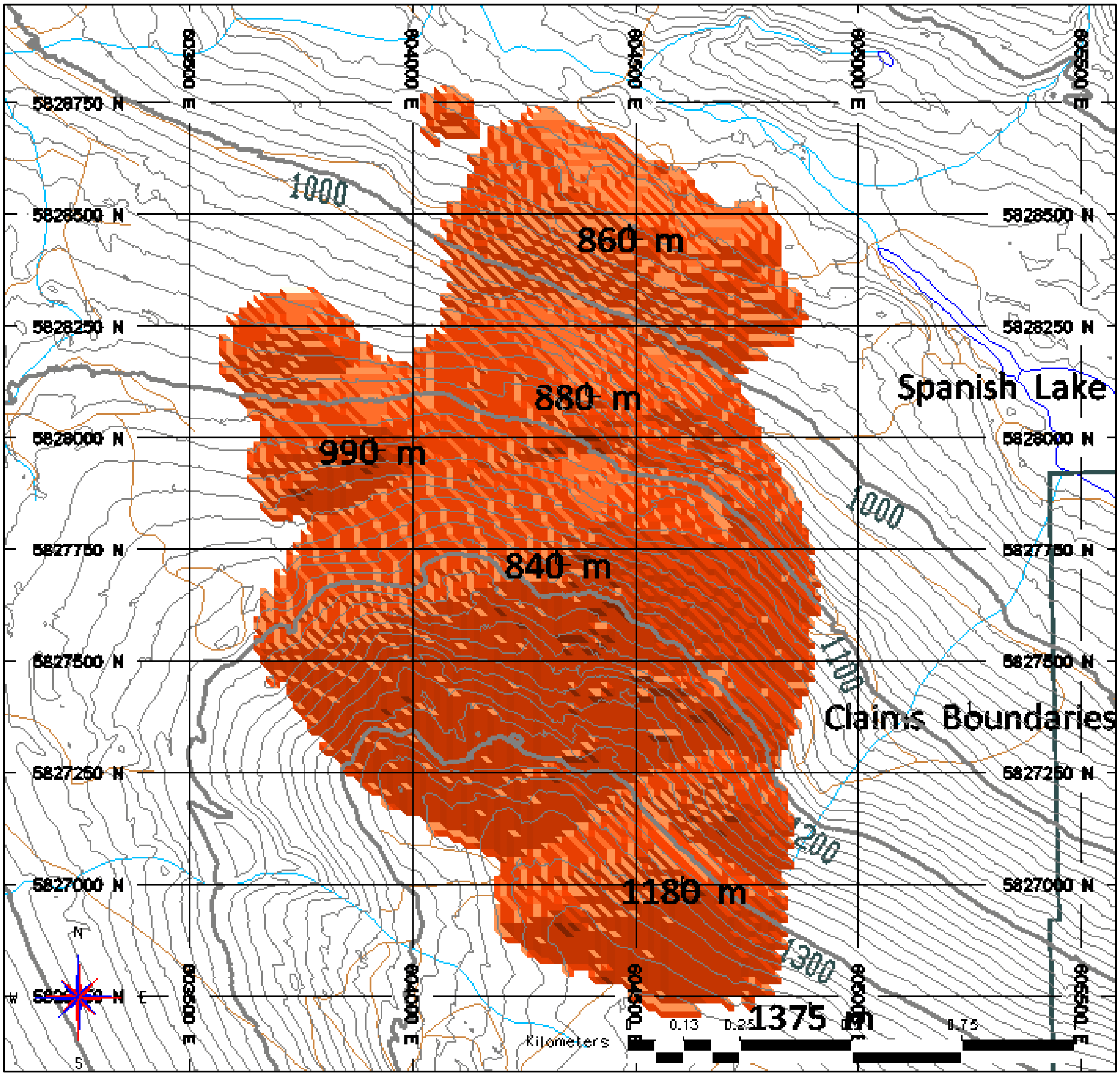
The pit shell generated at the 55% case is selected as the ultimate pit limit and is used for subsequent mine planning in this study. The Mineral Resources contained within this selected pit limit shell are shown in the Table 16-6 with a cut-off gold grade of 0.30 g/t. This shell target is used for further mine planning at Spanish Mountain Gold as a target for detailed open pit designs with berms and ramps.

Table 16-6 Pit Delineated Contents

Input Gold Price	$820	US$/oz
Measured and Indicated Resource	91,327	kt
Gold Grade	0.79	g/t
Waste and Inferred Resource	367,207	kt
Strip Ratio	4.0	Waste / Resource
Total Pit Contents	458,534	kt

Larger pit shells, developed at prices factor cases approaching 100%, provide future opportunity to economically develop the project beyond the chosen limits for this PFS.

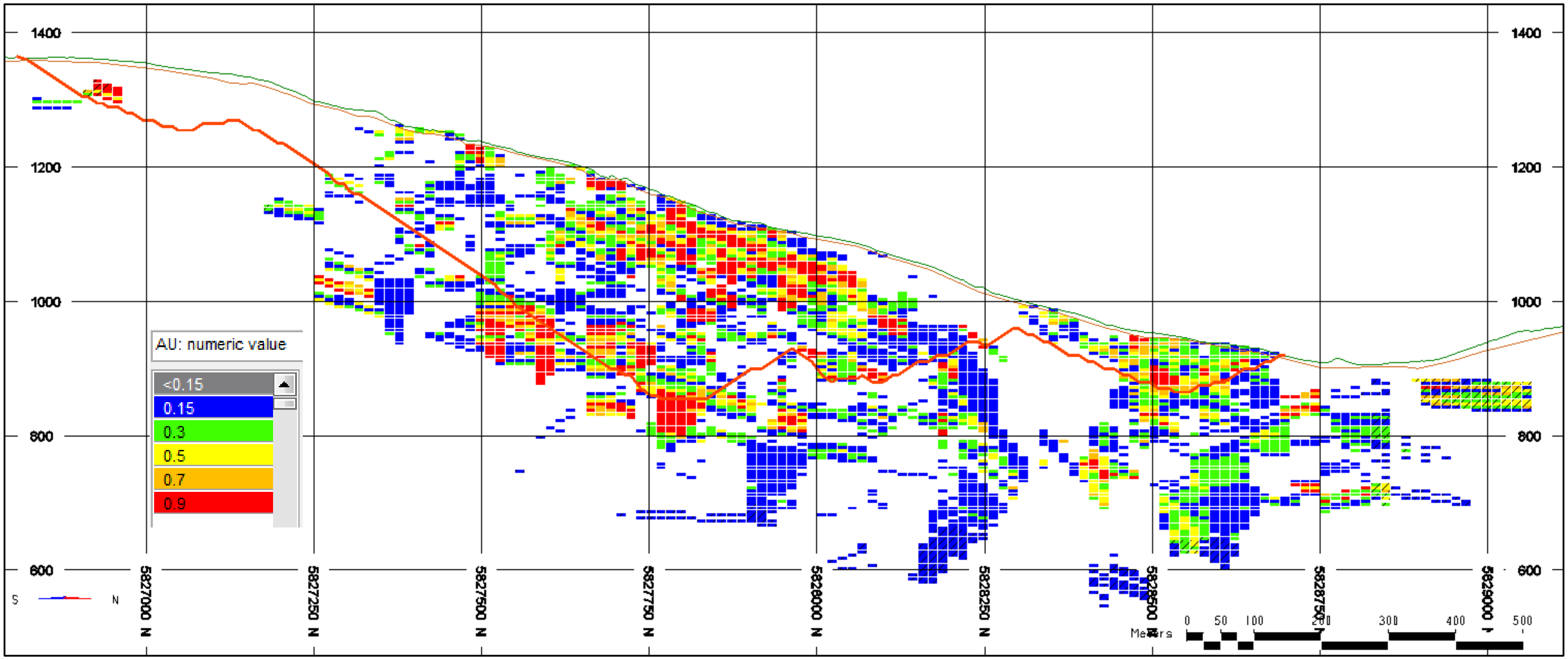
The following figures Figure 16-4 through Figure 16-6 show plan and section views of this chosen ultimate pit shell.



Source: Moose Mountain, 2021

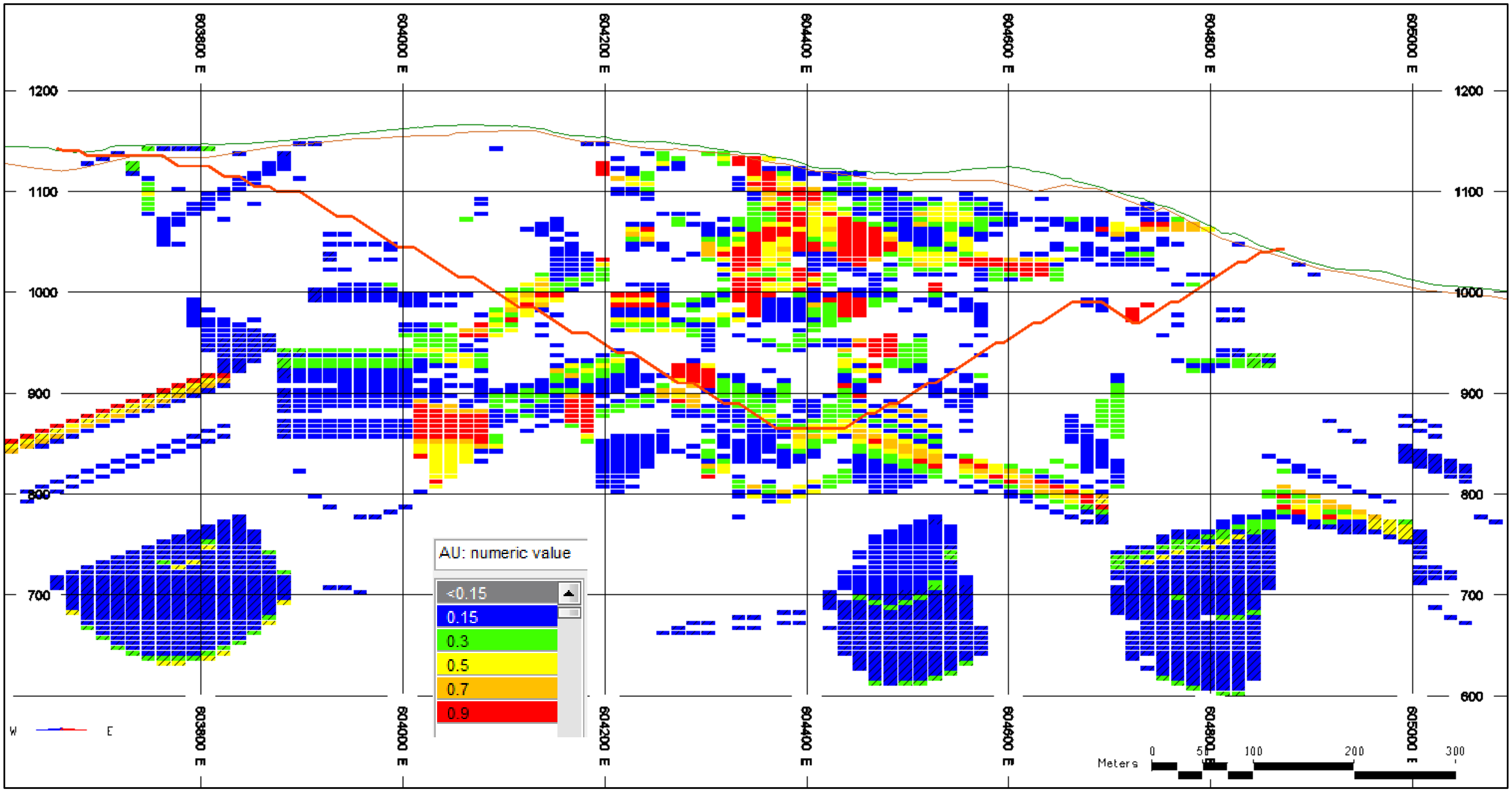
Figure 16-4 Plan View of Optimized Pit Shell

Block views show gold grade in all blocks above a 0.30 g/t cut-off. Mineralized block between 0.15-0.30 g/t gold are also shown for reference. Inferred class blocks are shown with hatching. Green line represents original topography, brown line represents the bedrock surface.



Source: Moose Mountain, 2021

Figure 16-5 Cross Section View, 604385E (looking west), of selected optimized pit shell



Source: Moose Mountain, 2021

Figure 16-6 Cross Section View, 5,827,850N (looking north), of selected optimized pit shell

16.4 Pit Phase Selection

The ultimate pit limits are generally split up into phases or pushbacks to target higher economic margin material earlier in the mine life and to even out strip ratio over the mine life.

Other considerations for selection of interim pit phases:

- Provide enough resource to sustain the plant operations for at least two years (14M tonnes).
- The pit benches should be large enough to allow an efficient area for mining and keep the vertical bench advance rate to be <9 benches per year.
- Minimum mining width to allow an efficient area for mining is assumed to be 60 m.

The pit optimization price cases described in the preceding sections can typically be used as a guideline for selecting interim pit phases. Pit shells created by the optimization algorithm with lower input gold prices than the selected ultimate pit case will contain higher grade resources and/or lower strip ratios.

The pit shell generated using a 31% price factor is used to target a starter pit phase. The next pit pushbacks target the 37% price factor pit shell, following by pushbacks to the 54% price factor pit shell. The final phases push out to the optimized pit limits in the south.

16.5 Pit Phase Designs

Pit designs are completed that demonstrate the viability of accessing and mining the potential resource. The designs are run with the following inputs:

- Variable bench heights, bench face angles, inter-ramp angles and overall wall angles based on the details in Table 16-4 and Figure 16-2.
- Suitable single and dual lane haul road widths to handle 140 t payload haulers.
    - 22 m for single lane traffic
    - 29 m for dual lane traffic
- The ramp is not extended into the bottom 10 m of the pit, assuming the ramp will be retreat-mined out of these benches.
- The ramp is designed to single- lane width for the 20 m above this, assuming single lane traffic is adequate.
- 10% maximum ramp grade.
- Pit exits face west towards the crusher, stockpiles, and tailings dam.

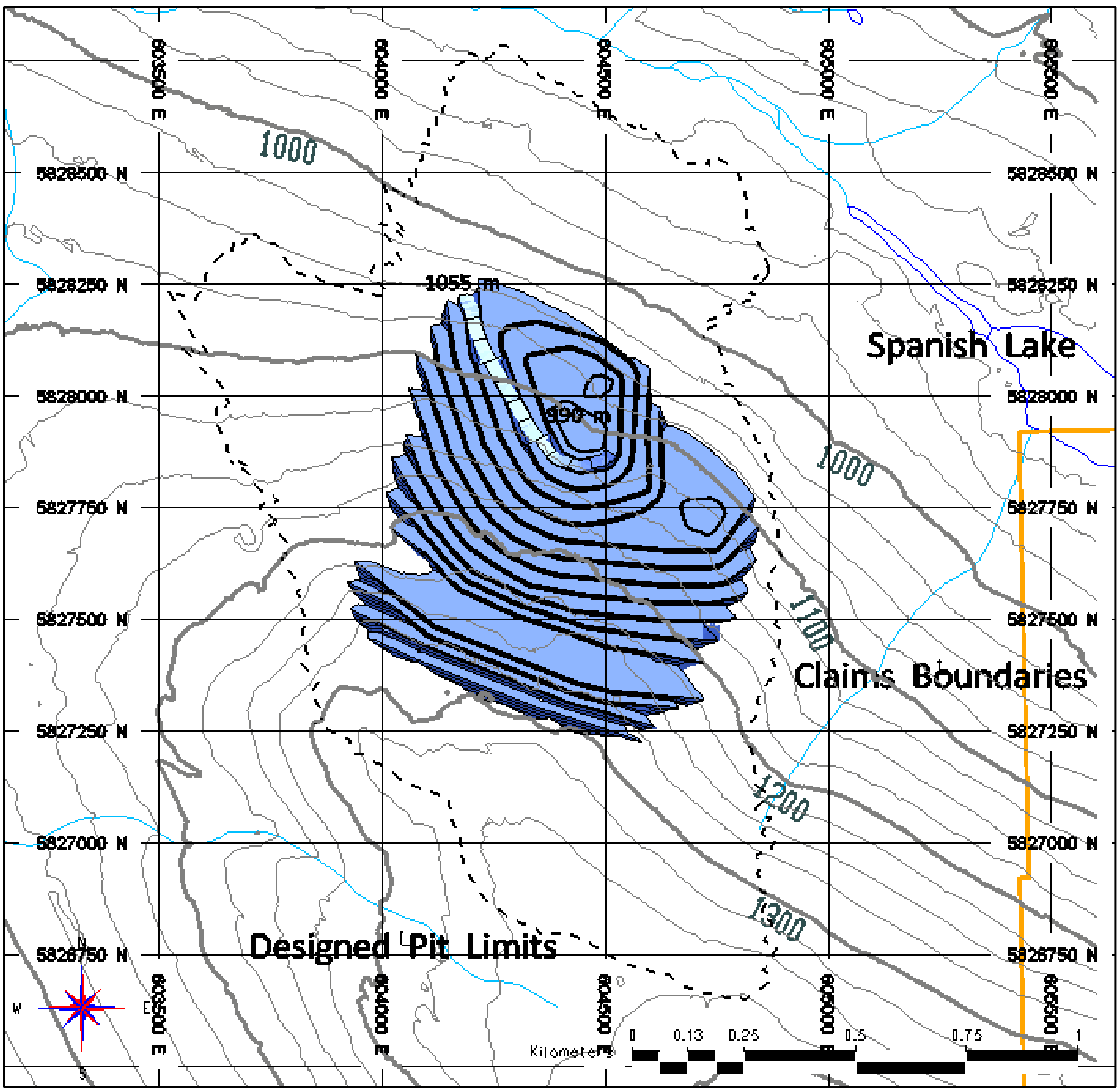
The following sections describe the designs of the open pit phases. The description of the detailed pit phase designs (or pushbacks) in this section uses the following naming conventions:

- The first digit signifies the type of geometry object (P6 is used for pit geometry).
- The middle digit signifies the design series.
- The final digit signifies the pit phase number.

The suffix 'i' indicates that the resource tonnage for the phase is incremental from the previous phase. If there is no 'i' specified, it is cumulative up to the phase indicated. The mining phases throughout the LOM are presented in Figure 16-7 through Figure 16-16.

16.5.1 Starter Pit, P621

This starter pit contains just over two years' worth of mill feed when utilizing an elevated cut-off gold grade. This phase is at a lower strip ratio and higher gold grade than the ultimate pit. This pit mines from the crest at the 1300 m elevation, down to the ramp exit on the 1,055 m elevation via external haul roads outside the pit limits and internal cut roads initially built within the pit limits. The ramp runs counterclockwise down to the pit bottom at the 990 m elevation.

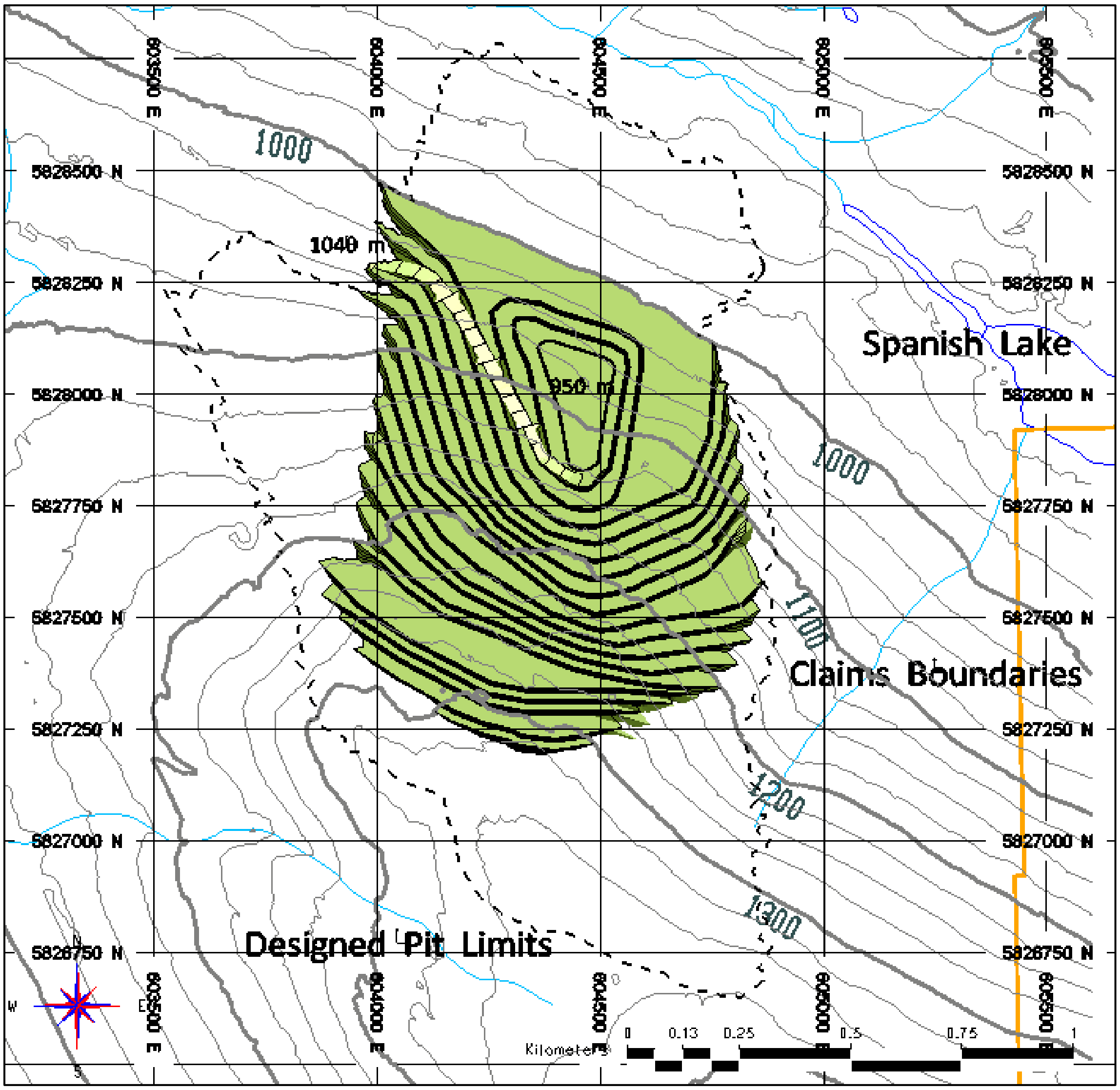


Source: Moose Mountain, 2021

Figure 16-7 Plan View of Starter Pit, P621

16.5.2 West Pushback, P622

P622 is a west pushback on the P621 pit. This pit mines from the pushback crest at 1220 m elevation down to the pit exit at the 1040 m elevation via external roads: then down the counterclockwise ramp to the pit bottom at the 950 m elevation.

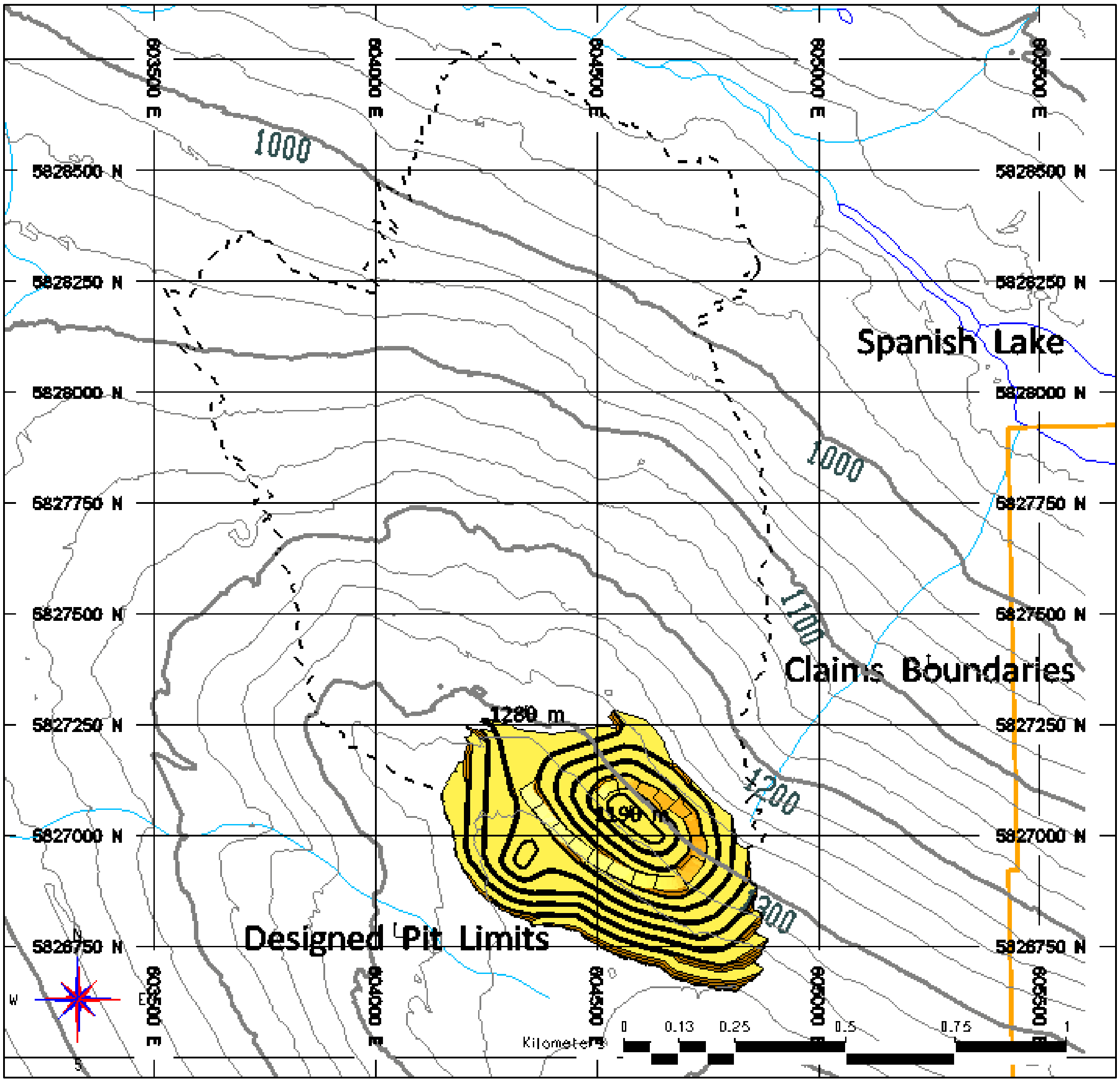


Source: Moose Mountain, 2021

Figure 16-8 West Pushback, P622

16.5.3 South Phase, P623

P623 is a standalone pit at the ultimate southeast side pit limits. This pit mines from the crest at 1380 m elevation down to the pit exit at the 1280 m elevation via external roads: then down the counterclockwise ramp to the pit bottom at the 1190 m elevation.

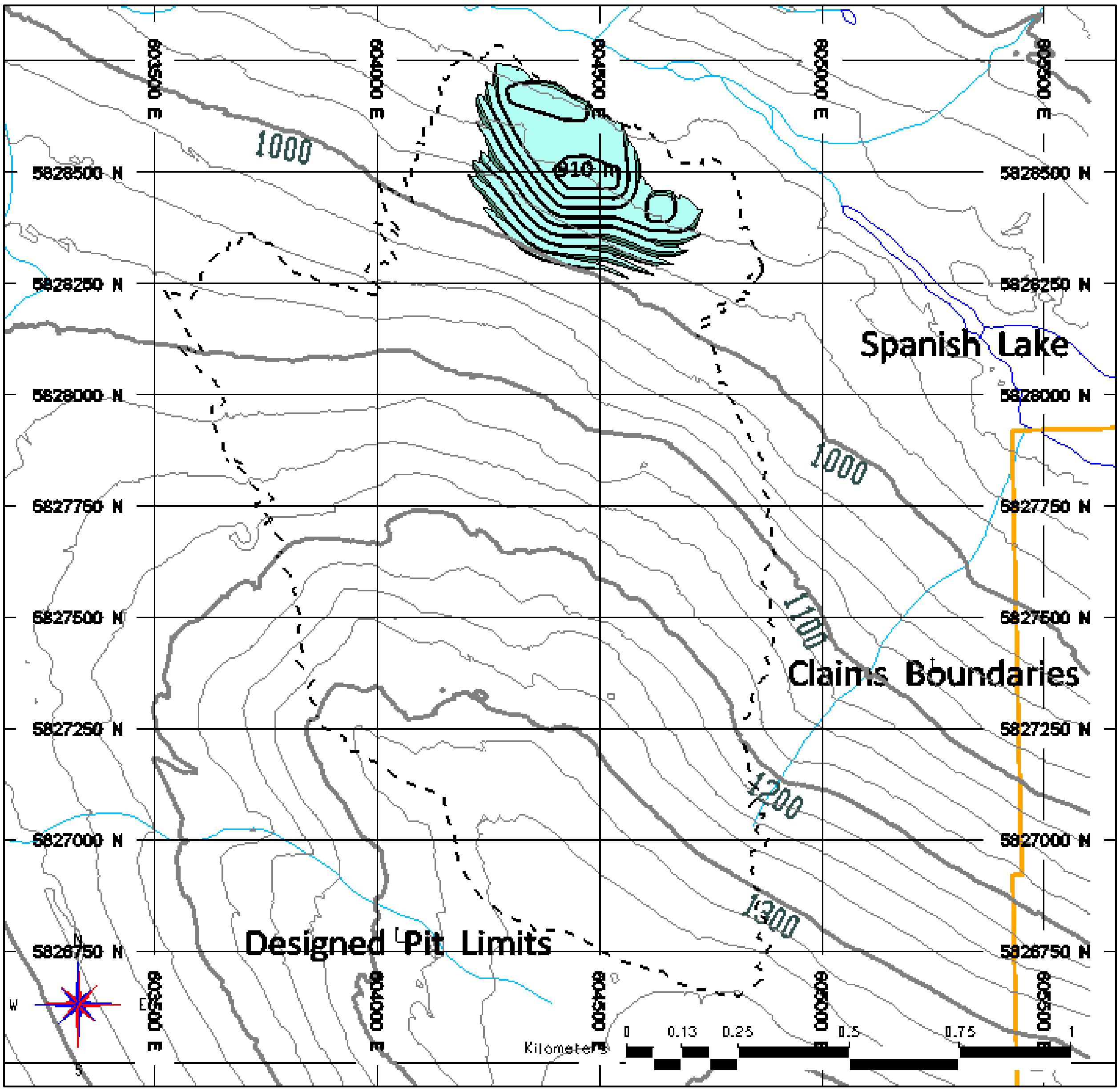


Source: Moose Mountain, 2021

Figure 16-9 South Phase, P623

16.5.4 North Starter Phase, P624

P624 is a starter pit for the north side of the Spanish Mountain Gold deposit. It is a standalone phase mined early in the mine life, with enough room to pushback to the ultimate limits in the northwest and northeast ends of the pit. This pit mines from the crest at 1000 m elevation down to the pit bottom at the 910 m elevation via external roads.

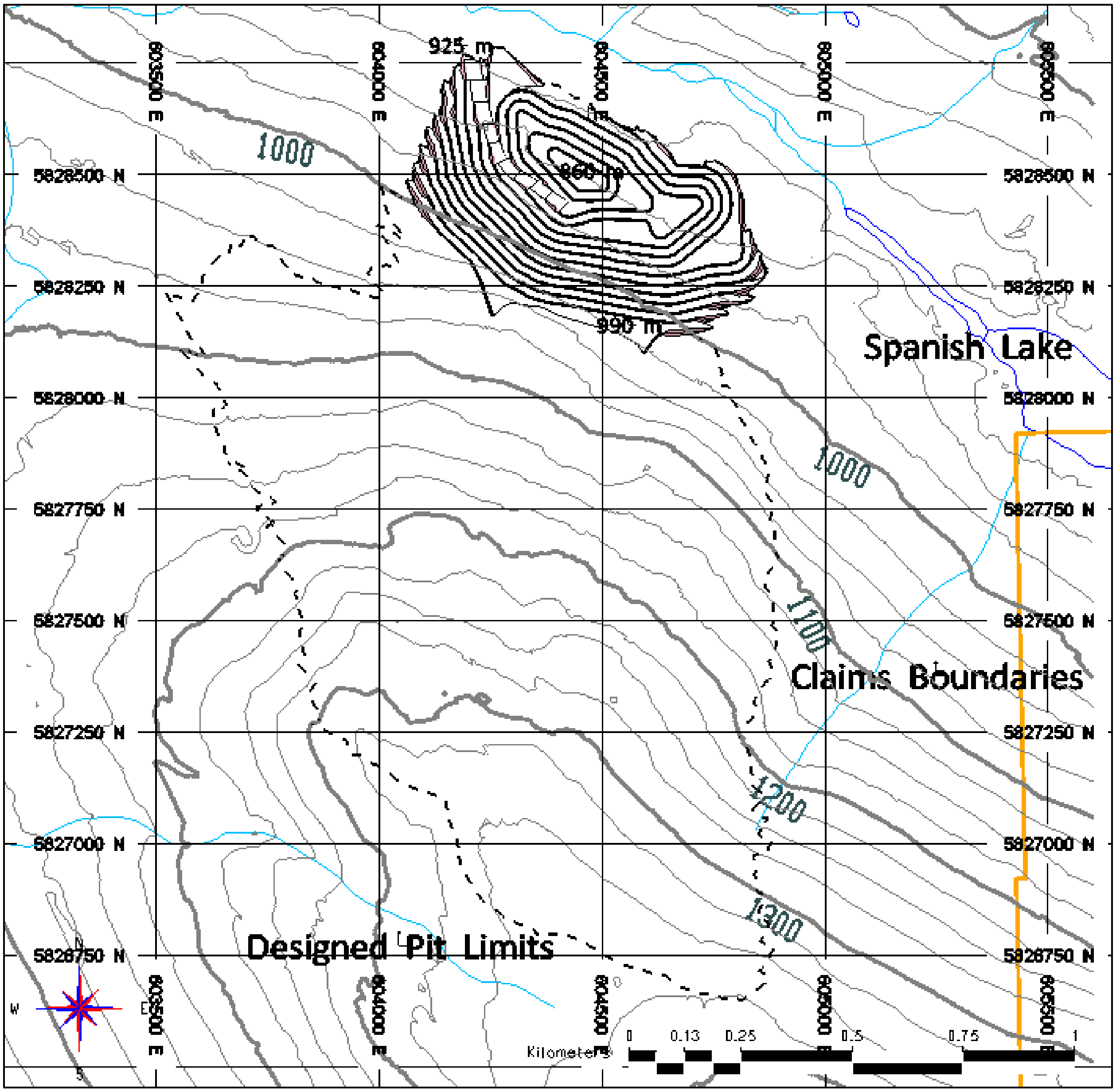


Source: Moose Mountain, 2021

Figure 16-10 North Starter Phase, P624

16.5.5 North Ultimate Phase, P625

P625 mines to the designed pit limits in the north portion of the deposit, pushing out from the previous phase. A saddle is created at the 990 m elevation between this phase and the phases to the south. This pit mines from the crest at 1000 m elevation (assuming P622 mines the bench above in advance of this phases development) down to the pit exit at the 925 m elevation via external roads; then down the counterclockwise ramp to the pit bottom at the 860 m elevation. Once mined out, this area can potentially be used for waste rock storage.

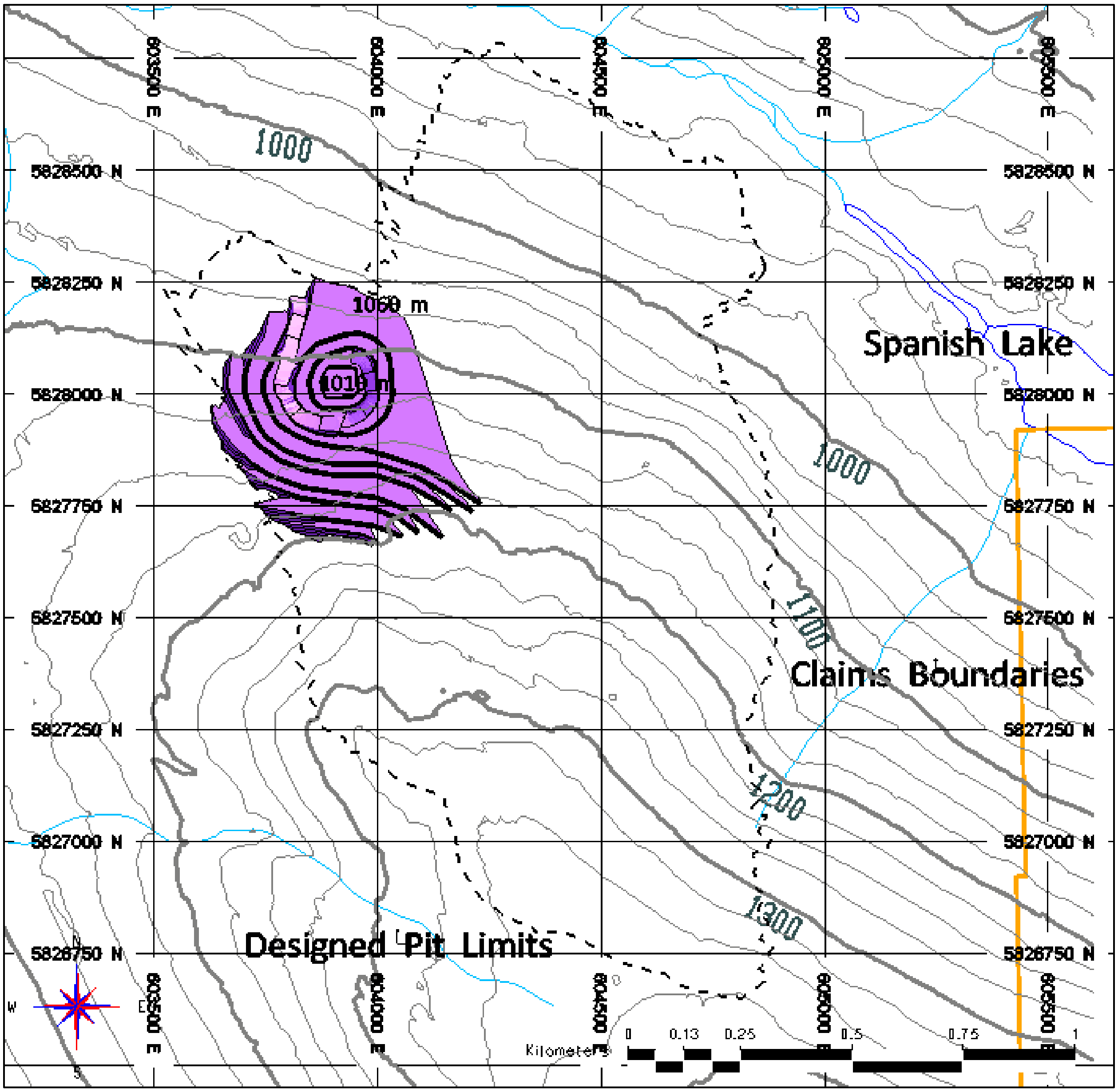


Source: Moose Mountain, 2021

Figure 16-11 North Ultimate Phase, P625

16.5.6 West Phase 1, P626

P626 pushes back the central pit to the west. This pit mines from the crest at 1200 m elevation, down to the pit exit at the 1070 m elevation via external roads: then down the counterclockwise ramp to the pit bottom at the 1010 m elevation.

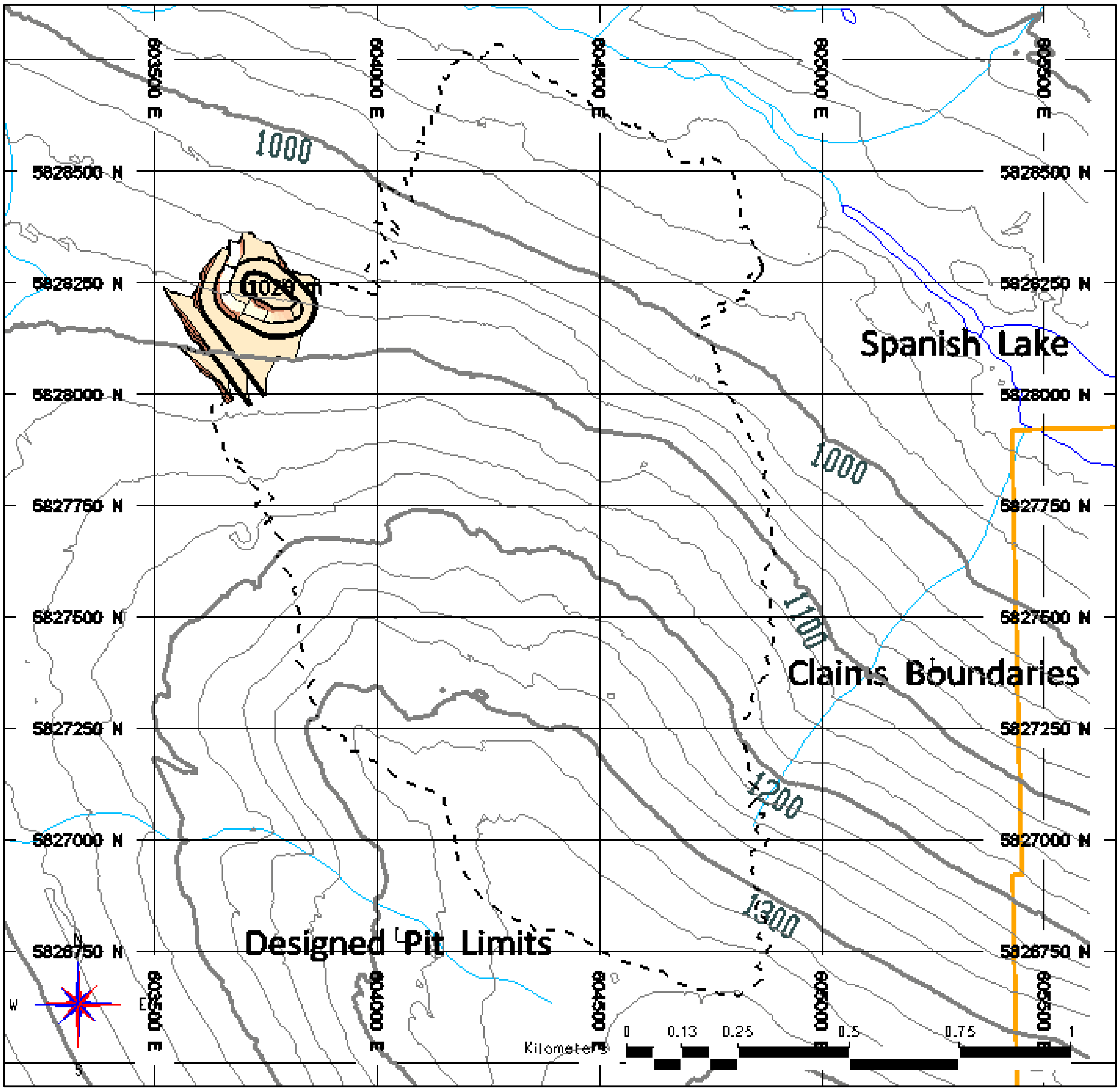


Source: Moose Mountain, 2021

Figure 16-12 West Phase 1, P626

16.5.7 West Phase 2, P627

P627 pushes back the previous phase to the ultimate west pit limits. This pit mines from the crest at 1120 m elevation, down to the pit exit at the 1055 m elevation via external roads: then down the counterclockwise ramp to the pit bottom at the 1020 m elevation.

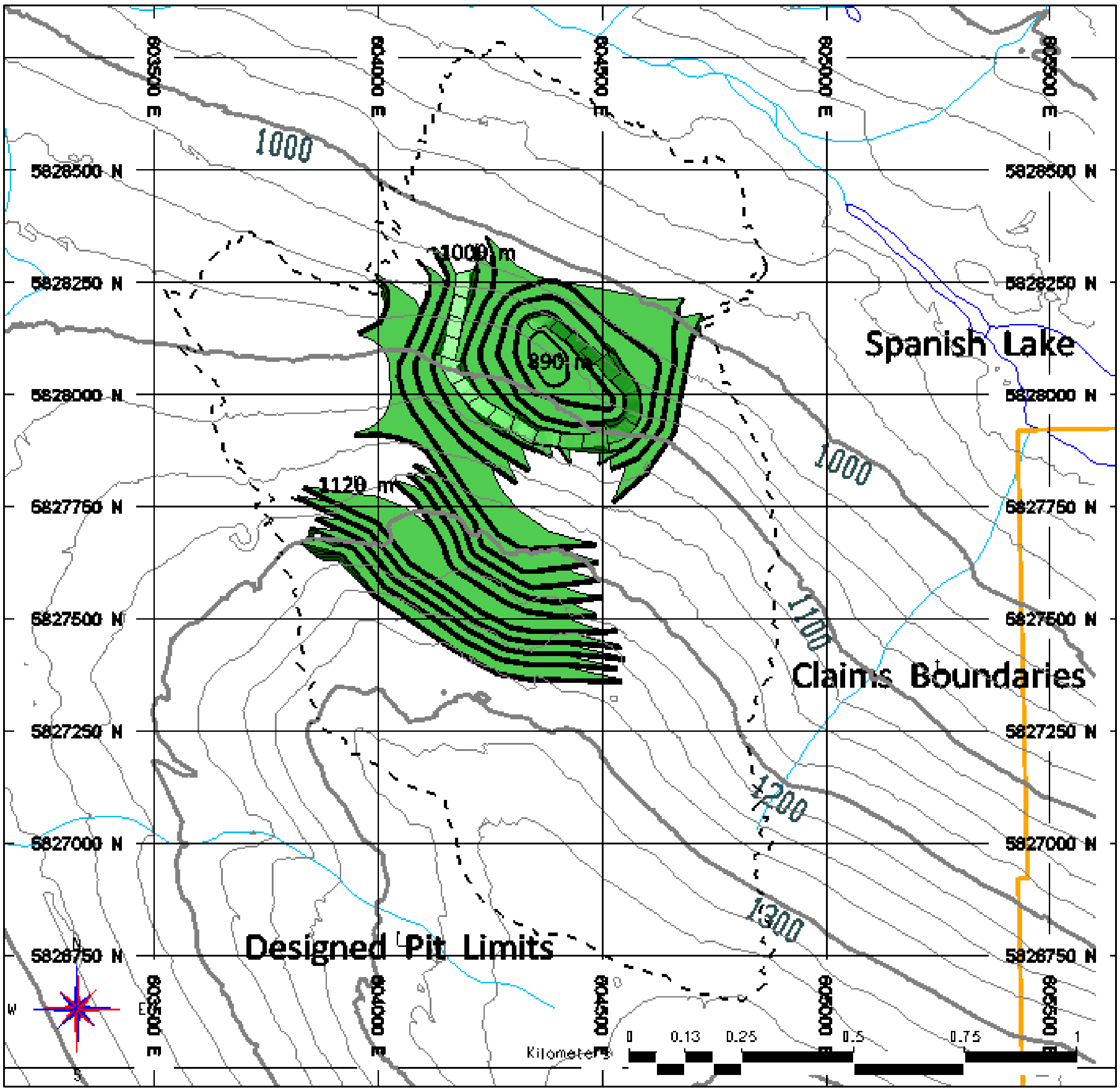


Source: Moose Mountain, 2021

Figure 16-13 West Phase 2, P627

16.5.8 South Pushback 1, P628

P6228 pushes the central portion of the pit out to the south, mining from the pushback crest at 1220 m elevation down to the pit exit at the 1000 m elevation via external roads: then down the counterclockwise ramp to the pit bottom at the 890 m elevation. This phases development assumes that the preceding phases have mined the benches above the 1220 m elevation.

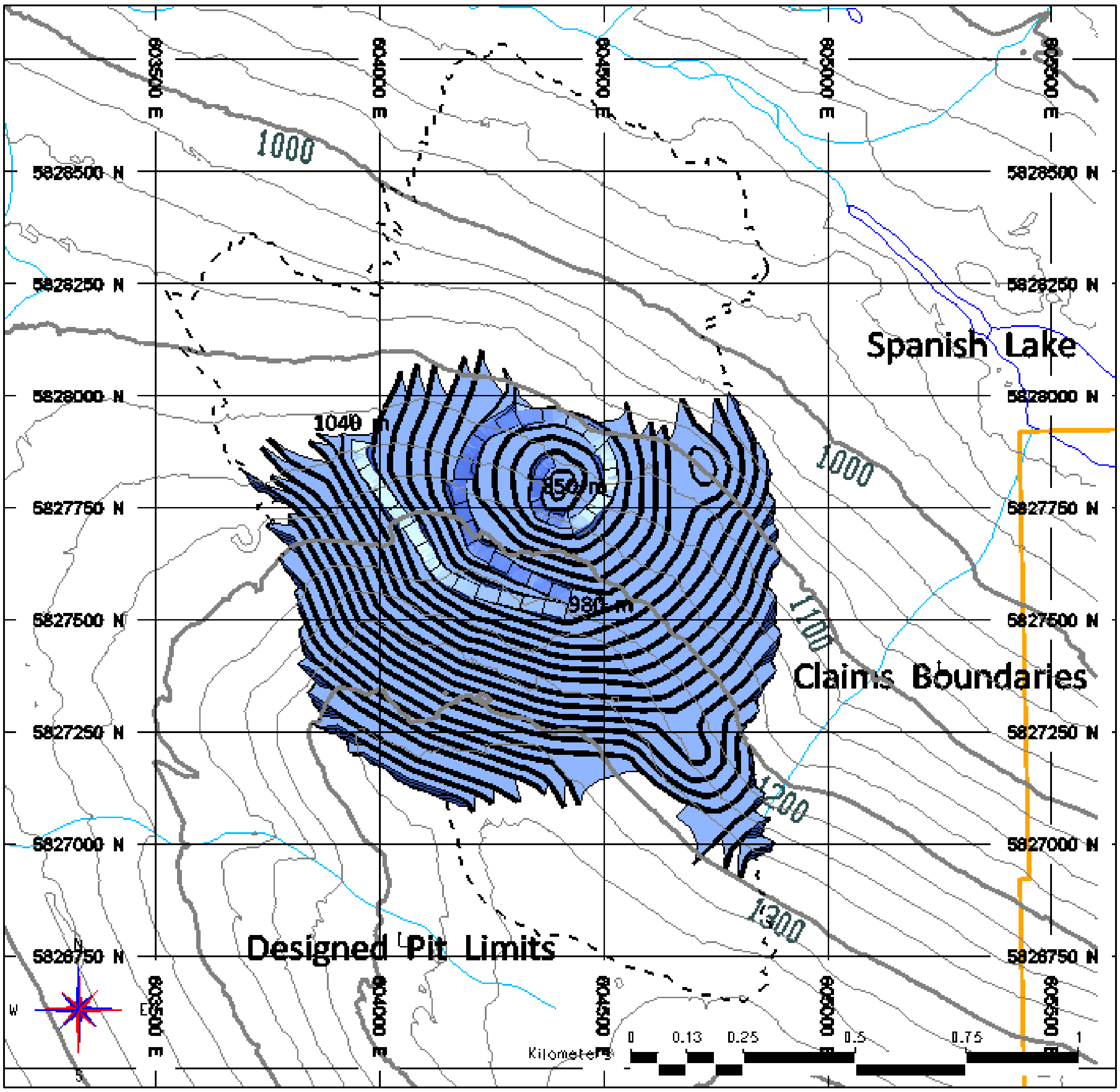


Source: Moose Mountain, 2021

Figure 16-14 South Pushback 1, P628

16.5.9 South Pushback 2, P629

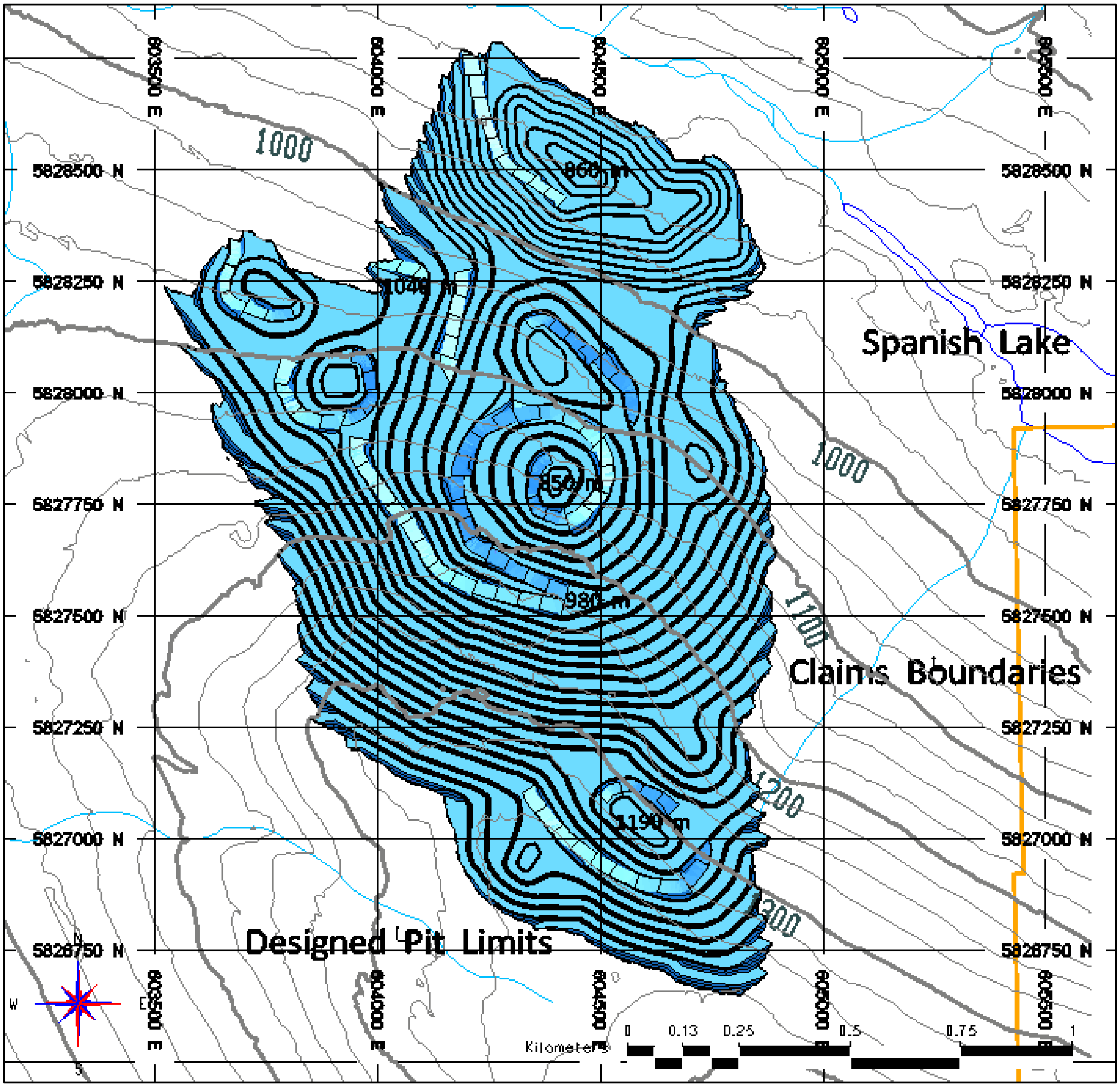
P629 pushes the pit out the ultimate limits in the south. This pit mines from the crest at 1340 m elevation down to the pit exit at the 1040 m elevation via external roads: then down the counterclockwise ramp, switch backing at the 980 m elevation and down to the pit bottom at the 850 m elevation. P623 precedes this phase, mining the benches above.



Source: Moose Mountain, 2021

Figure 16-15 South Pushback 2, P629

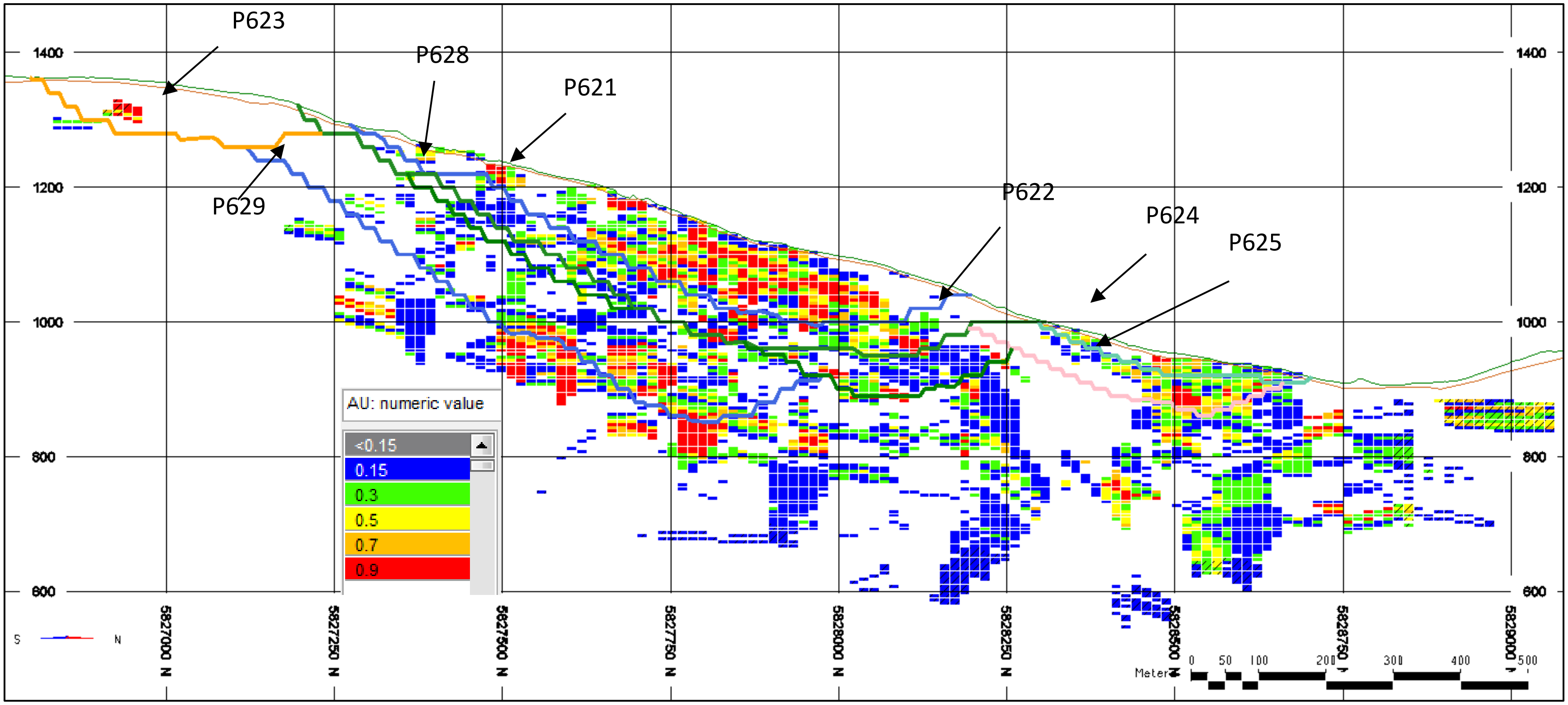
16.5.10 Ultimate Pit for Mine Planning



Source: Moose Mountain, 2021

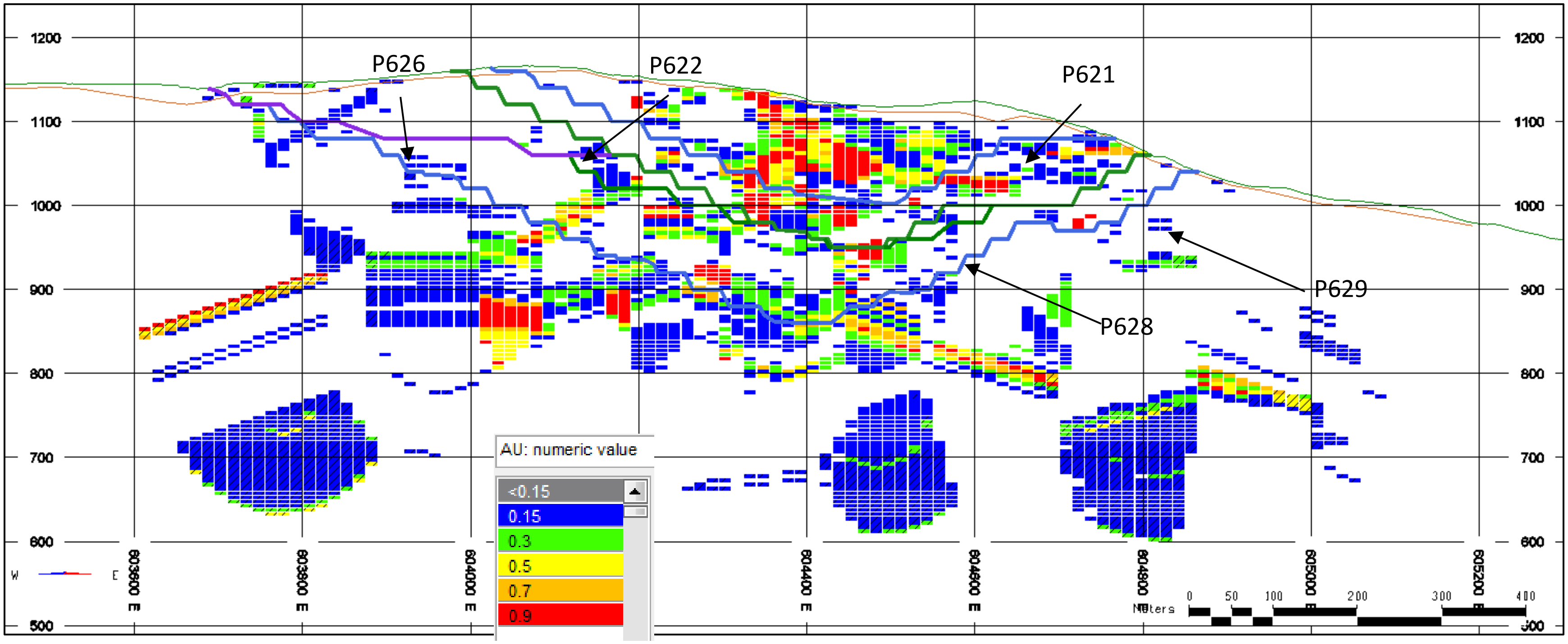
Figure 16-16 Plan View of Ultimate Pit for mine planning

Block views show gold grade in all blocks above a 0.30 g/t cut-off. Mineralized block between 0.15-0.30 g/t gold are also shown for reference. Inferred class blocks are shown with hatching. Green line represents original topography, brown line represents the bedrock surface.



Source: Moose Mountain, 2021

Figure 16-17 Cross Section View, 604385E (looking west) of Phased Pit Designs

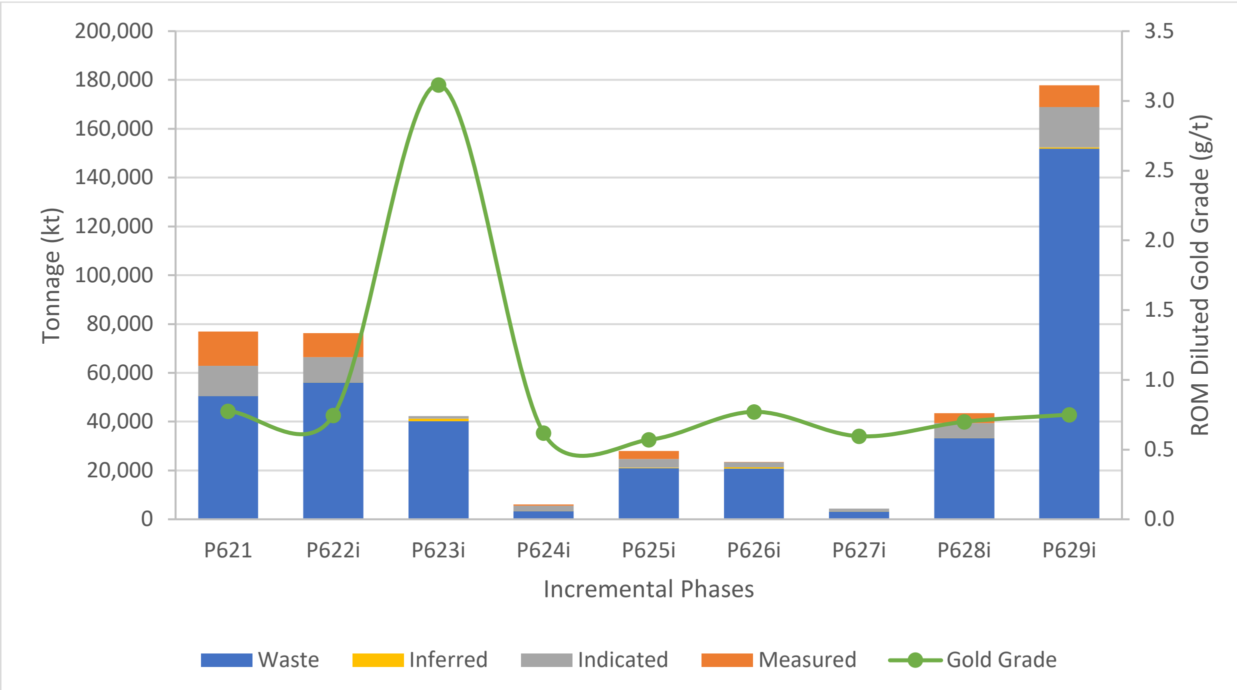


Source: Moose Mountain, 2021

Figure 16-18 Cross Section View, 5,827,850N (looking north), of Phased Pit Designs

16.6 Pit Contents

The Mineral Reserves delineated by the pit designs are shown in Table 15-2 and illustrated in Figure 16-19. The utilized cut-off gold grade is 0.30 g/t, and Inferred Mineral Resources are treated as waste rock.



Source: Moose Mountain, 2021

Figure 16-19 Designed Phase Pit Contents

16.7 Waste Rock Management

16.7.1 Waste Rock Characterization

Acid rock drainage (ARD) potential criteria in the pit waste rock was formalized in a technical memo by SRK Consulting (Canada) Inc. (SRK, 2012) to Spanish Mountain Gold in 2012. The pertinent formulas and criteria to categorize the pit waste rock is summarised below. Sulphur, calcium, and arsenic values were interpolated into the 3DBM to quantify the acid rock drainage (ARD) generating potential of the pit material.

The ARD classification is defined as:

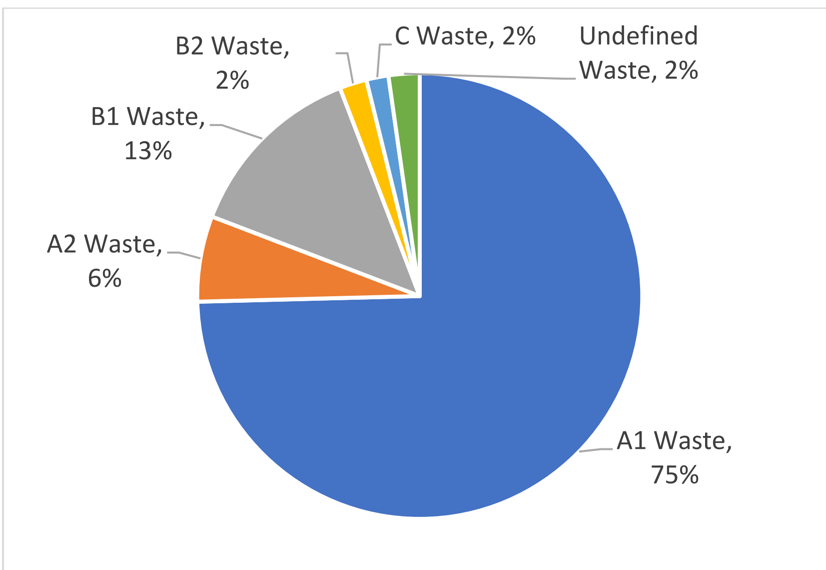
- Acid Potential (AP) for block = 31.25 X S
- Neutralization Potential (NP) for block = 37 X Ca + 8.8,

where S is the sulphur value in percent, and Ca is the calcium value in percent. The ARD categories are defined as follows:

- Ai: if NP/AP ratio > 2 and arsenic < 150 ppm; unlikely to generate ARD and low potential for arsenic leaching – i.e., unlikely to require management.
- Aii: if NP/AP ratio > 2 and arsenic > 150; unlikely to generate ARD, arsenic leaching potentially significant.
- Bi: if NP/AP ratio > 1 and <= 2, and arsenic < 150; unlikely to generate ARD and low potential for arsenic leaching – i.e., unlikely to require management.
- Bii: if NP/AP ratio > 1 and <= 2 and arsenic > 150; same as Bi for ARD, but arsenic leaching potentially significant.
- C: if NP/AP ratio <= 1 for all arsenic values; PAG, very likely to require management.

These categories were written into the 3DBM (item ARD) to characterize each block.

Figure 16-20 shows the distribution of the waste rock ARD categories contained in the ultimate pit. Most waste is Ai (75%) and Bi (13%), which is categorized as non-acid generating potential. The undefined category indicates that values for either, or all, sulphur, calcium, or arsenic are not interpolated into the model blocks, likely due to missing data.



Source: Moose Mountain, 2021

Figure 16-20 Pit Waste Rock Distribution by ARD Characterization

16.7.2 Waste Rock Suitability

The possible waste rock destinations are determined by their ARD generating characterization. There are three main destinations:

- the waste rock storage facilities (WRSF) on surface
- the tailings dam and water management pond embankments
- the tailings pond for sub-aqueous disposal.

The destinations for each of the ARD categories are as follows:

- Ai: Non ARD, can be placed anywhere (e.g., WRSF's or tailings embankment).
- Aii: Potential ARD, requires management (e.g., sub-aqueous placement within no specific time frame, and can be used for upstream dam construction).
- Bi: Non ARD but placed sub-aqueously within the tailings facility.
- Bii: Potential ARD, requires management – sub-aqueous placement after one year and assumed that it will be directed immediately to the tailings pond and co-mingled.
- C: ARD, required to be sub-aqueously placed immediately – will go to the tailings pond and co-mingled.

It is assumed that 75% of the undefined category of waste rock will have the same characteristic as Ai categorized material, while the remaining 25% will be like Aii.

16.7.3 Waste Rock Disposal Strategy

Suitable mine waste rock, Ai category, will be hauled from the pit and placed on two external WRSF's, North WRSF and West WRSF, both on the west side of the pit.

A placed waste density of 2.3 t/m^3 is assumed for all waste rock, and 2.1 t/m^3 for overburden, and is used to size the potential areas for waste rock storage. Initial layouts for the WRSF's use an overall 2.8:1 slope (19.6 degrees) from crest to toe, built from 15 m lift heights dumped out at angle of repose.

All overburden waste will be segregated into a stockpile directly west of the open pit. The final elevation at the top of the overburden stockpile will be 1,160 m, containing 9 Mt of overburden.

Waste rock material from the initial years and upper mining benches will be hauled to the West WRSF. It will be constructed by a combination of staged lifts and wrap- arounds. Access from the pit will be from roads constructed along the contours at strategic elevations to maintain level or downhill hauls where possible. The final elevation at the top of the WRSF will be 1,320 m, containing 93 Mt of waste rock.

The North WRSF will be built with the waste rock from mining benches at the north end of the pit, as well as from the lower elevations. This WRSF design is physically constrained by Spanish Creek to the north and Hepburn Lake to the northwest. The plant site is immediately to the south restricting its advancement in that direction. The top elevation will be 1,080 m, containing 79 Mt of waste rock.

Though there is also available space nearby for disposing of waste rock on the east side of the pit, it is not considered for this study due to potential impacts on the drainages in that direction.

Suitable mine waste rock, Ai category, will also be hauled to the tailing's facility and water management ponds for dam embankment construction, as required. It is estimated that 21 Mt of material will be required from the pit for both the north and south dam embankments, as well as the water management pond through the LOM; including 6.5 Mt during the pre-production period. Limited samples from test pits indicate that the overburden is unconsolidated with high moisture content and may not be competent for dam embankment construction. Overburden, as well as undefined materials, are therefore not considered for dam embankment construction.

Waste rock that requires management and cannot be placed in the WRSF's or used for construction of the tailings embankment will be placed in the tailings pond for subaqueous disposal. The ARD categories for this pit waste include Aii, Bi, Bii, and C. A total of 72 Mt of waste rock has been planned for subaqueous disposal in the tailings pond. Some Aii and waste rock will be used for upstream tailings embankment construction, where it is assumed that the material will be submersed in less than two years.

Mineralized waste rock (between 0.15 and 0.30 g/t gold or inferred classification) has been segregated in the mine plan to report to a WRSF west of the pit and south of the plant site. The top elevation of this sub-grade WRSF pile is 1,170 m, with a capacity of 77 Mt. The mine plan does not reclaim this material back to the mill, but future studies should examine the opportunity to mill this material after the open pit has been exhausted.

All Inferred class resources have been treated as waste rock, but for the purposes of this study, have been disposed of within the low grade mineralized WRSF described above.

The layout for the WRSF's can be seen Figure 1-4.

16.8 Ex-pit Haul Roads

Ex-pit haul roads are designed with a maximum grade of 10%. A 35 m width that incorporates a dual-lane running width and berms on both edges of the haul road is sized to accommodate 140-tonne payload class haulers. The roads will be built from blasted waste rock fill from the pits, topped with a crushed rock running surface. The costs to construct these ex-pit haul roads are assumed to be accounted for in the costs to haul and dump waste rock to the WRSF, with additional costs of crush rock production and handling added.

Initial development to the top benches of the starter pit phase will be constructed from cut internal to the pit limits. The 35-m wide roadway will be cut up the side of the hill, the retreat mined out to the surrounding ex-pit haul roads.

Ex-pit haul road layouts can be seen in Figure 1-4.

16.9 Ore Stockpiles

When ore is mined from the pit, it will either be delivered to the crusher, the ROM stockpile located next to the crusher, or the ore stockpiles.

A cut-off grade strategy has been employed for the production schedule, and during operations a stockpile near the crusher will be maintained to store these reserves for later re-handle back the crusher.

A "High-Grade (HG)" stockpile is built to the south of the crusher. This stockpile is planned to be reclaimed to the crusher once the open pit operations are completed. The top elevation of this stockpile is 1,110 m, with a capacity of 7 Mt.

Ore stockpile layouts can be seen in Figure 1-4.

16.10 Mine Production Schedule

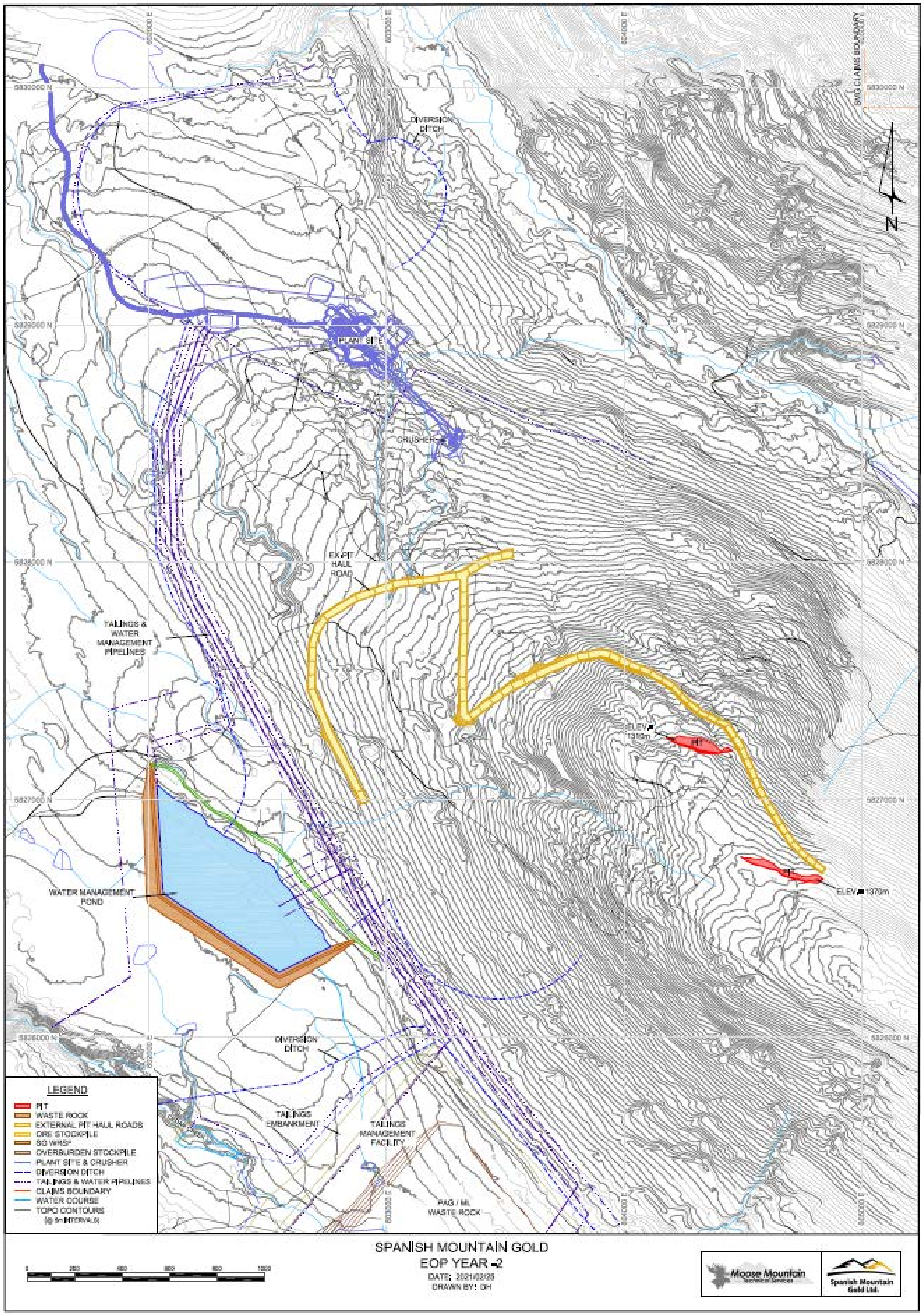
16.10.1 Pre-Production Development

During the pre-production period, mine related activities will include site clearing, stripping and stockpiling topsoil, establishing perimeter ditches, and haul road development. Approximately 10 km of haul roads must be developed during the pre-production period to access the top benches of the pit from the crusher and stockpile, and from the open pit to the tailing's facility and WRSF's.

A pre-strip of 12 Mt of is required. The construction of the starter tailings dam and water management pond will require 6.5 Mt during the pre-strip. Haul roads will require an additional 2.0 Mt during the pre-strip. The remainder is associated resource material that will be stockpiled, 0.2 Mt, overburden material that will be stockpiled, 1.2 Mt, and PAG rock that will be stockpiled within the tailings facility footprint, 2.2 Mt.

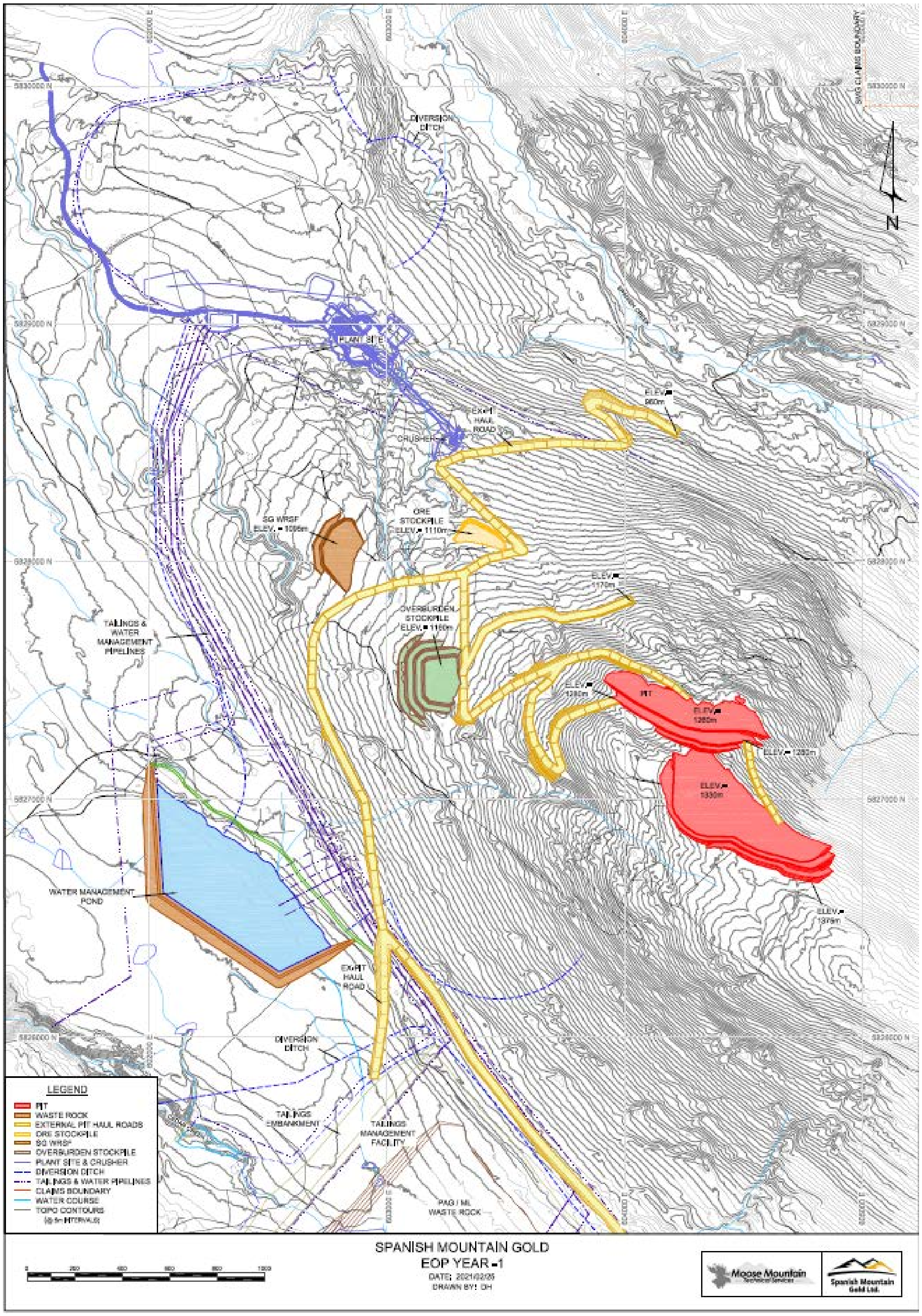
The top several benches in the Starter pit phase will be developed with a pioneering equipment fleet consisting of a small diameter track drill and dozers until a workable mining bench can be established for larger, more productive equipment. It is anticipated that this work will be carried out by a contract miner who will also construct the initial mine haul roads prior to the owner's mine equipment fleet being available.

Figure 16-21 and Figure 16-22 illustrate the mining activities to be completed by the end of the pre-production period, at the end of Y-2 and start of mill operations, respectively. The water management pond, and other site infrastructure, is shown in these figures for reference, but timing for construction of these items may not match the mine operations sequence shown.



Source: Moose Mountain, 2021

Figure 16-21 Mine Development - Pre-Production, Y-2



Source: Moose Mountain, 2021

Figure 16-22 Mine Development - Pre-Production, Y-1

16.10.2 Production Scheduling

The open pit mine production schedule is based on the following parameters:

- Annual mill feed of 7,300,000 t is targeted based on an average of 20,000 tonnes/day milling.
  - Mill production will ramp up through Year 1, totalling 6,000,000 t.
- Periods are scheduled quarterly through the construction (pre-production) period and the first two years of mill operations, then annually for the remaining mine life.
- Phased pit bench resources and waste rock contents are used as input to the mine production schedule.
- Within a given phase, each bench is fully mined before progressing to the next bench. Optimization by partial bench mining is not examined at this level of study, even in zones of predominately waste rock.
- Pit phases are mined sequentially, with more than one phase in production in a period. A subsequent phase is limited from progressing vertically beyond its predecessor.
- Pit phase progression is limited to no more than nine benches each year. Average phase progression in any annual period is five 10 m benches.
- A cut-off grade strategy is employed. An elevated gold cut-off grade of 0.40 g/t is employed during the early years of the open pit operations.
- Material in the stockpiles is reclaimed to the mill after the pit is mined out in Year 13.

The mine production schedule is shown in the following Table 16-7 through Table 16-9 and Figure 16-23 and Figure 16-24.

Table 16-7 Mine Production Schedule

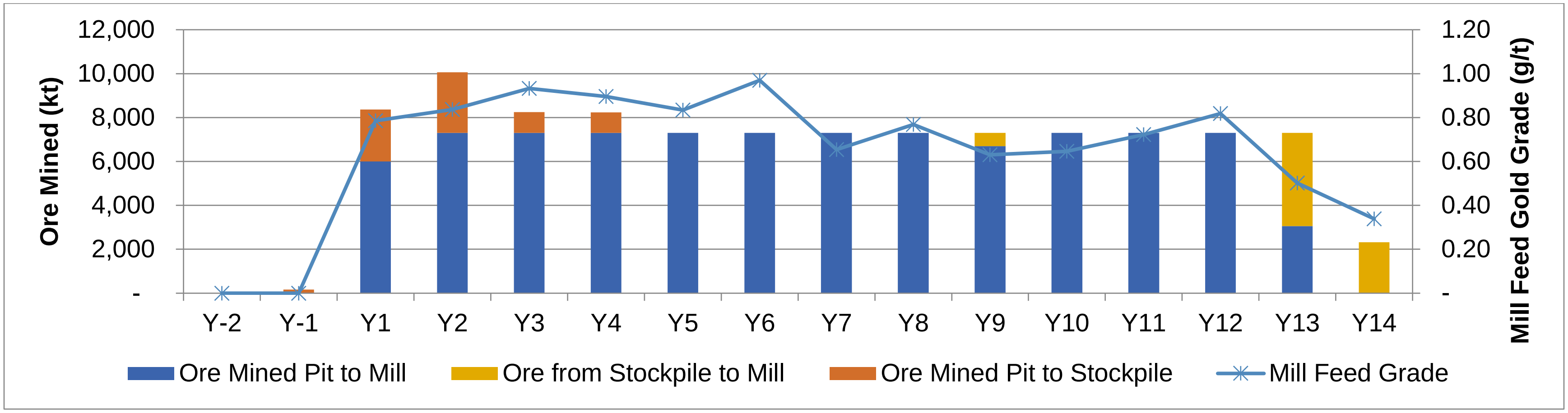
Period	Pit to Mill (kt)	Pit to Stockpile (kt)	Stockpile to Mill (kt)	Total to Mill (kt)	Au Feed Grade (g/t)	Ag Feed Grade (g/t)	Waste Rock (kt)	Strip Ratio (t:t)	Total Mined (kt)	Total Moved (kt)
Y-2							891		891	891
Y-1		162					11,845	73.0	12,007	12,007
Y1	6,000	2,363		6,000	0.79	0.82	23,185	2.8	31,548	31,548
Y2	7,300	2,758		7,300	0.84	0.70	24,349	2.4	34,407	34,407
Y3	7,300	949		7,300	0.93	0.73	24,409	3.0	32,658	32,658
Y4	7,300	934		7,300	0.90	0.66	26,713	3.2	34,946	34,946
Y5	7,300			7,300	0.83	0.71	27,423	3.8	34,723	34,723
Y6	7,300			7,300	0.97	0.74	41,795	5.7	49,095	49,095
Y7	7,300			7,300	0.65	0.71	40,722	5.6	48,022	48,022
Y8	7,300			7,300	0.77	0.70	40,700	5.6	48,000	48,000
Y9	6,700		600	7,300	0.63	0.67	41,300	6.2	48,000	48,600
Y10	7,300			7,300	0.65	0.71	30,665	4.2	37,965	37,965
Y11	7,300			7,300	0.72	0.70	31,598	4.3	38,899	38,899
Y12	7,300			7,300	0.82	0.69	13,611	1.9	20,912	20,912
Y13	3,059		4,241	7,300	0.50	0.69	3,702	1.2	6,760	11,001
Y14			2,325	2,325	0.34	0.70				2,325
Total	**88,760**	**7,166**	**7,166**	**95,927**	**0.76**	**0.71**	**382,907**	**3.7**	**478,834**	**486,000**

Table 16-8 Pit to Mill of Stockpile Production Schedule, Phase Details

Period	Ex-Pit Const. Strip	P621, Starter Pit	P622, West Pushback	P623, South Pushback	P624, North Starter Pit	P625, North Ultimate Pit	P626, West Pit 1	P627, West Pit 2	P628, South Pushback 1	P649, South Pushback 2	Total (kt)
Y-2	725		142	24							**891**
Y-1		1,115	2,903	7,988							**12,007**
Y1		16,516	8,926		6,107						**31,548**
Y2		32,797		1,610							**34,407**
Y3		17,582	6,202	6,967						1,907	**32,658**
Y4		8,493	14,750	11,703							**34,946**
Y5			21,436	13,288							**34,723**
Y6			18,105	673		4,340	4,873		4,887	16,217	**49,095**
Y7			3,780			10,797	16,438	3,238	7,105	6,664	**48,022**
Y8						11,078	2,088	1,091	39	33,704	**48,000**
Y9						1,752			11,509	34,739	**48,000**
Y10									19,464	18,501	**37,965**
Y11									519	38,379	**38,899**
Y12										20,912	**20,912**
Y13										6,760	**6,760**
Total	**725**	**76,503**	**76,245**	**42,252**	**6,107**	**27,967**	**23,399**	**4,329**	**43,523**	**177,783**	**478,834**

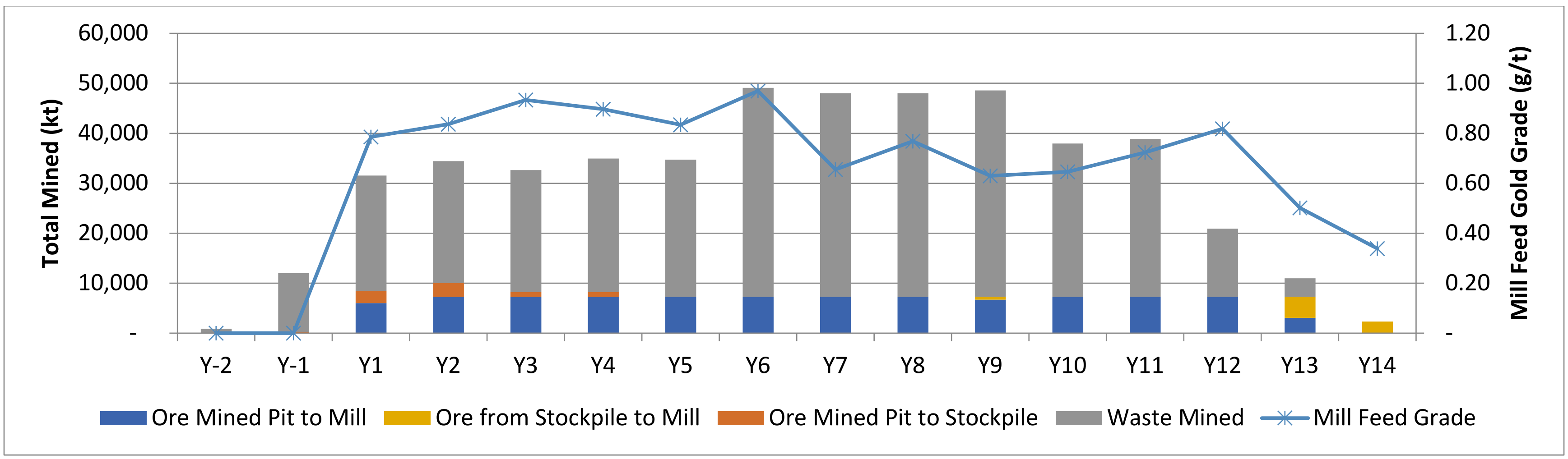
Table 16-9 Quantities by Non Crusher Destination

Period	Haul Road Rock Fill (kt)	Tailings Dam Embankment (kt)	Sub-Aqueous Disposal (kt)	Water Management Pond (kt)	North Waste Rock Stockpile (kt)	West Waste Rock Stockpile (kt)	Backfill Rock Stockpile (kt)	Sub Grade Waste (0.15 to 0.30 g/t gold) Stockpile (kt)	Inferred Stockpile (kt)	High Grade (> 0.30 g/t gold) Stockpile (kt)
Y-2	123		41							
Y-1	1,947	4,554	2,175	1,936				281	190	162
Y1	2,000		5,325	1,028		7,517		5,509	163	2,363
Y2		1,125	6,507			7,967		7,467	169	2,758
Y3		1,175	6,745		1,000	8,914		5,400	163	949
Y4		1,125	5,633		1,000	14,218		4,025	286	934
Y5		1,125	5,156		5,200	9,316		4,799	1,184	
Y6		1,125	7,995		8,700	16,516		5,429	550	
Y7		1,125	8,456		14,100	8,240		6,434	1,250	
Y8		1,125	6,652		15,000	11,243		4,796	1,375	
Y9		1,125	7,302		17,300	8,734		6,630	28	
Y10		1,125	4,757		16,686			7,930	104	
Y11		1,125	3,481				19,905	6,901	135	
Y12		1,125	959				7,054	4,473		
Y13		1,125	518				330	1,729		
Total	**4,070**	**18,104**	**71,702**	**2,964**	**78,986**	**92,663**	**27,289**	**71,803**	**5,599**	**7,166**



Source: Moose Mountain, 2021

Figure 16-23 Mine Production Schedule, Resource Mined and Mill Feed Grades

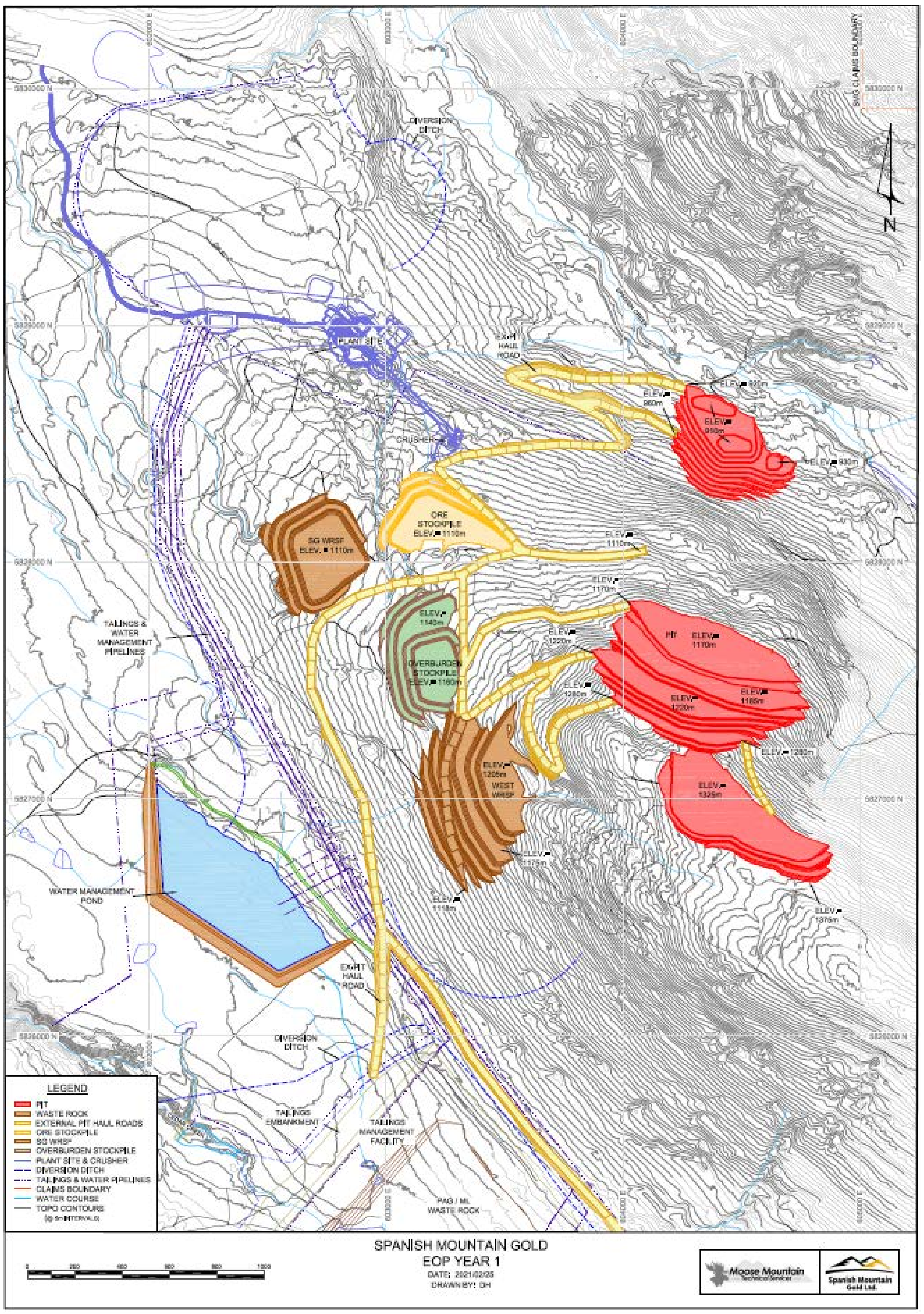


Source: Moose Mountain, 2021

Figure 16-24 Mine Production Schedule, Total Mined and Strip Ratio

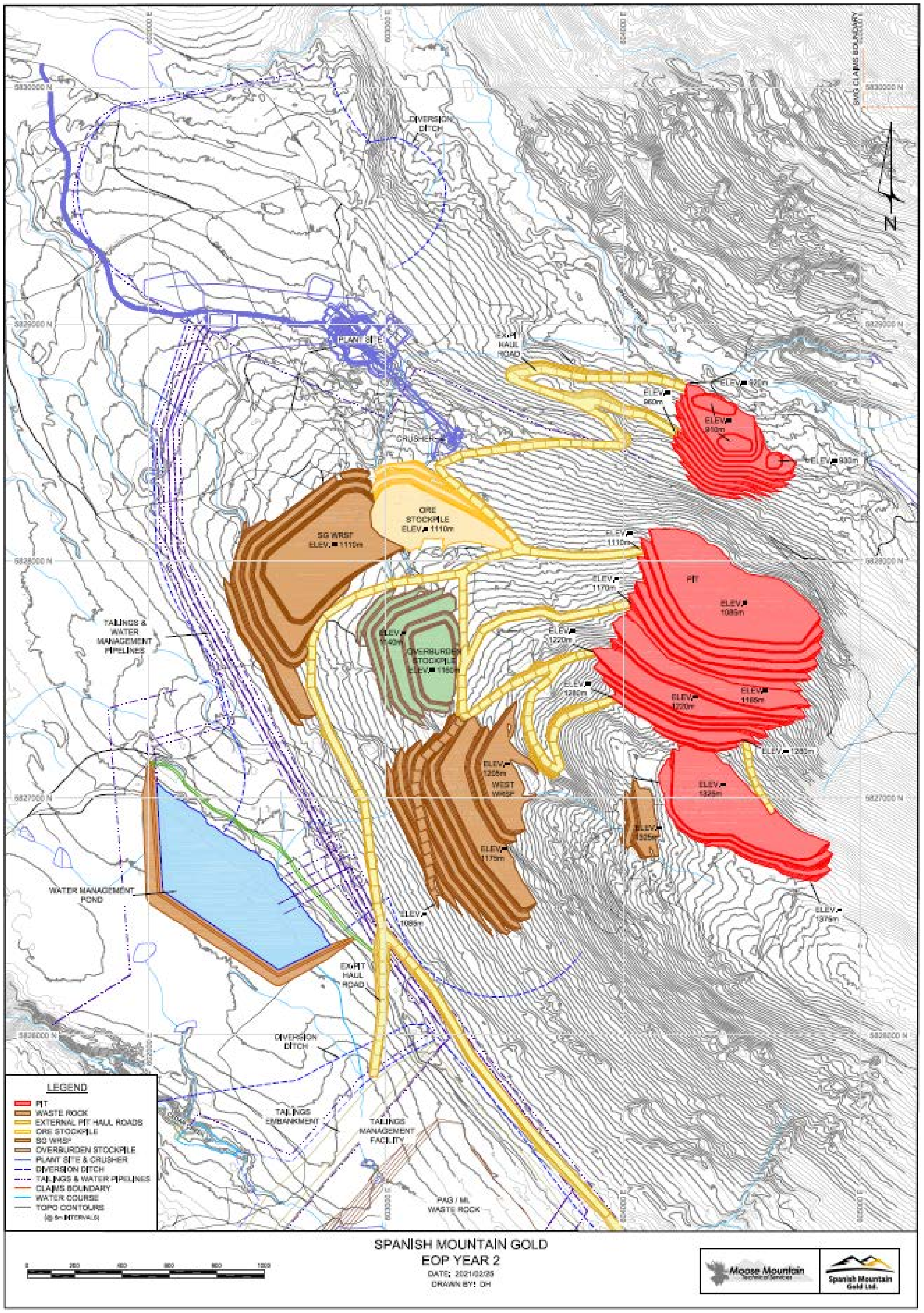
16.11 Mine End of Period Maps

The following Figure 16-25 through Figure 16-29 show the general arrangement of the mine operations at Year 1, Year 2, Year 5, Year 8 and Year 14, the completion of open pit operations.



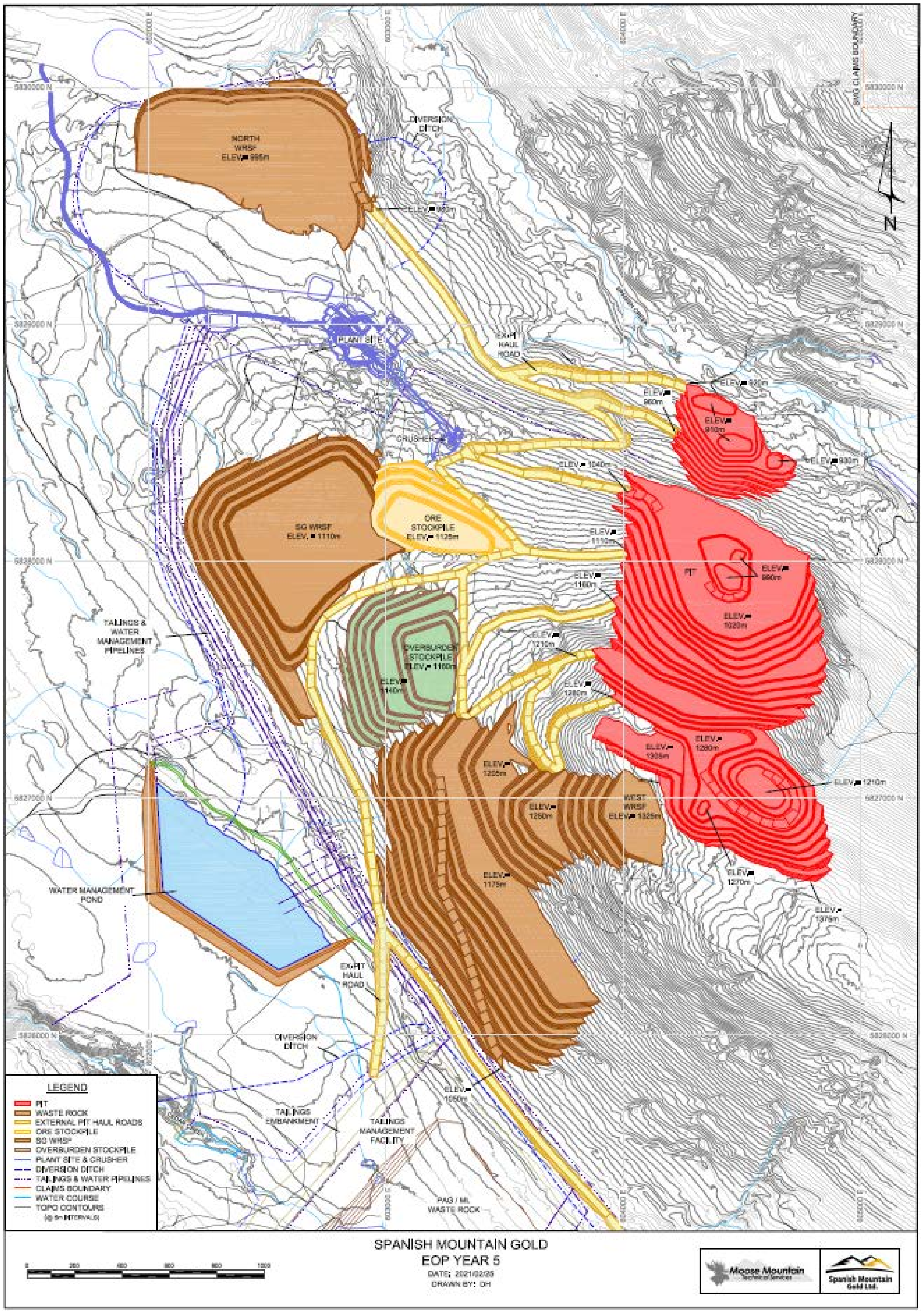
Source: Moose Mountain, 2021

Figure 16-25 Year 1 End of Period Map



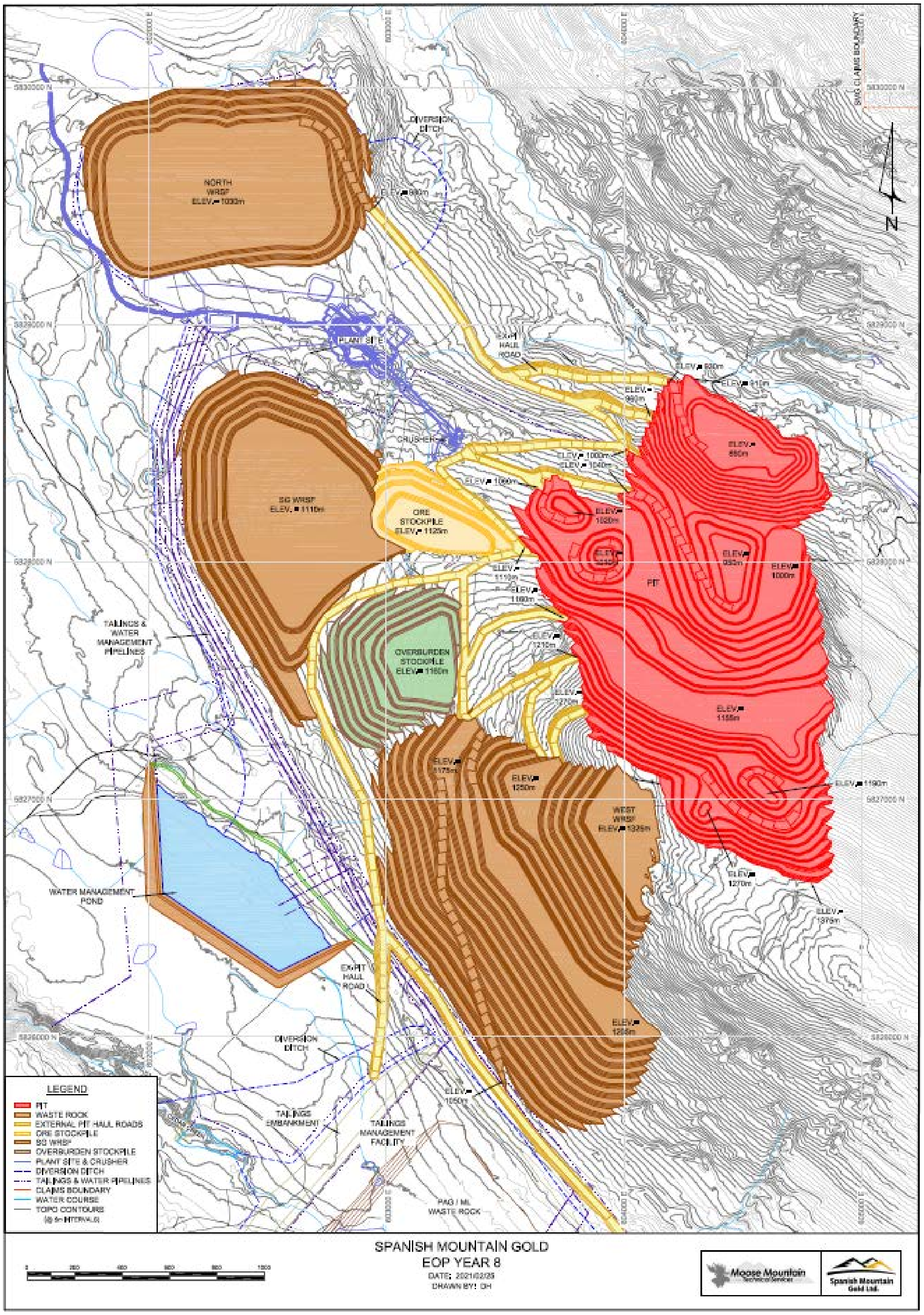
Source: Moose Mountain, 2021

Figure 16-26 Year 2 End of Period Map



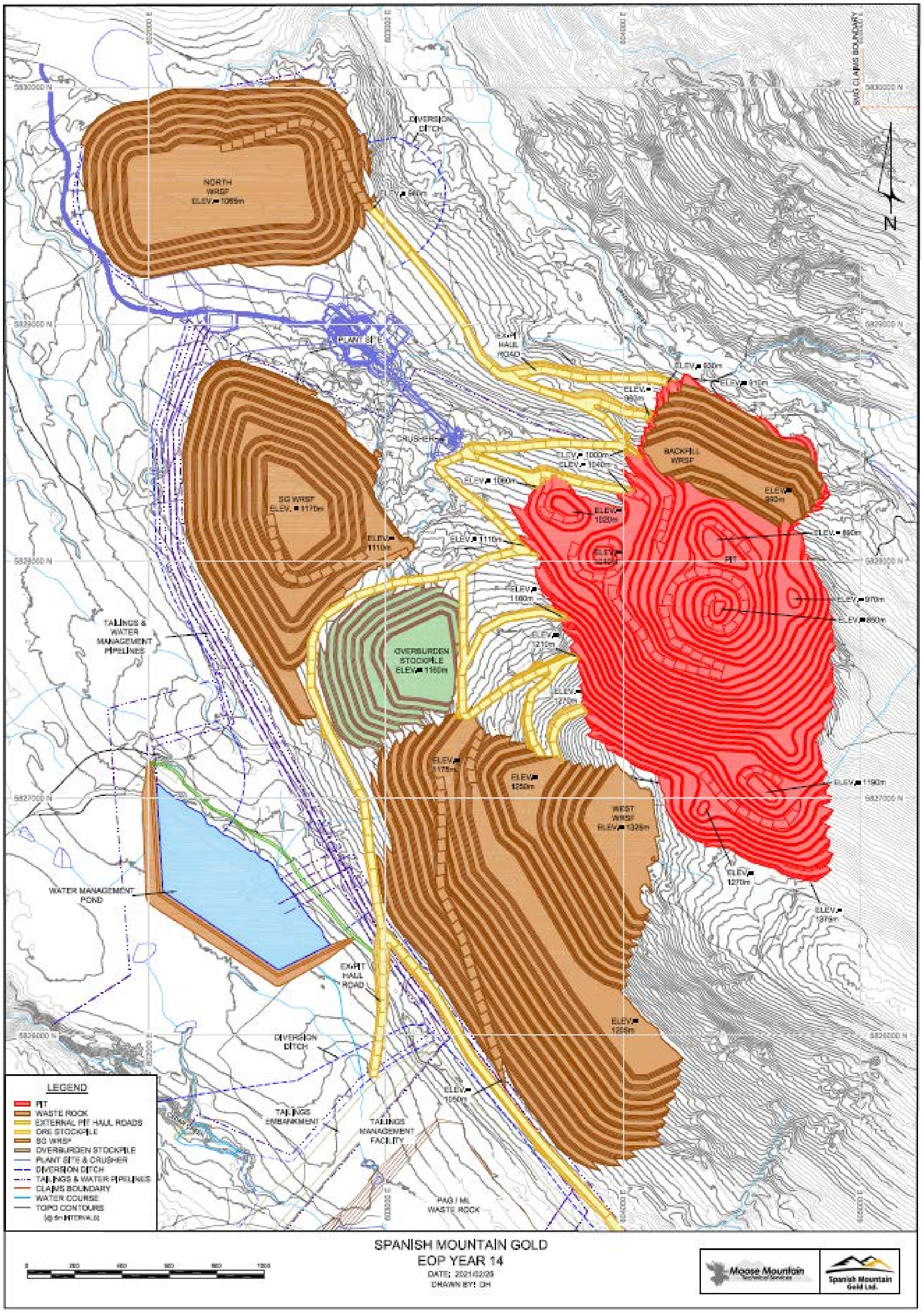
Source: Moose Mountain, 2021

Figure 16-27 Year 5 End of Period Map



Source: Moose Mountain, 2021

Figure 16-28 Year 8 End of Period Map



Source: Moose Mountain, 2021

Figure 16-29 LOM End of Period Map

16.12 Mine Operations

The mining operations are planned to be typical of similar scale open pit operations in mountainous terrain.

The mine fleet consists of the mobile equipment operating from the pit to the primary crusher, and to the WRSF's. It is assumed that the mine equipment fleet will be available on-site by Q2 of Year -2. The operating cost of the mine operation during the pre-production period has been included in initial capital and includes pre-stripping. Development work required prior to then will be undertaken by a contractor employing its own equipment fleet. Pit electrification will not be required as all equipment will be diesel powered.

Insitu rock is drilled and blasted to create suitable fragmentation for efficient loading and hauling of both waste rock and ore. A drill and blast plan has been scoped out to provide a powder factor to produce particle size distribution and diggability suitable for high productivity from the selected loader and haul truck fleet. Ore and waste rock is defined in the blasted muck pile with a grade control system based on grade control drilling, modeling of dig boundaries on smaller intervals, dig limit digitalization to the loader operators, field demarcation, and blast movement monitoring. A fleet management system keeps track of each load and ensures delivery to the correct destination. Mining benches will be 10 m high, with a 5 m split bench where a higher degree of mining selectivity is necessary.

Mine Operations are organized into two areas, Direct Mining and General Mine Expense (GME).

16.12.1 Direct Mining

Direct Mining includes the equipment operating costs, and operating labour, for the Grade Control, Drilling, Blasting, Loading, Hauling, Pit Support and Ground Support activities in the mine. Each section accounts for all equipment consumables and parts, manpower required (both operating and maintenance) and all operating supplies. This also includes the distributed mine maintenance items such as maintenance labour and repair parts, operation of mine maintenance equipment and tooling, plus off-site repairs which contribute to the hourly operating cost of the equipment.

16.12.2 Drilling

Diesel powered rotary drills capable of drilling 200 mm diameter holes will be used for production drilling. Four drills will be required. Pit wall control will be established using trim blasting techniques with a diesel powered DTH drill capable of drilling 127 mm holes. Further studies on rock structures and quality will determine whether pre-shearing with smaller diameter drillholes will be effective for highwall control.

16.12.3 Blasting

A target powder factor of 0.25 kg/t is proposed for the drilling and blasting operations. A contract explosives supplier will provide the blasting materials and technology for the mine, as well as a crew for blasting operations. A mixed emulsion type of explosive is assumed, with a higher ratio of emulsion to ANFO (ammonium nitrate and fuel oil) assumed in wet holes.

Blasting explosives will be manufactured off-site, and the on-site explosives bulk storage will be housed in a secure structure. The facilities will be located away from the mill site, pit, and all working areas, in compliance with regulatory requirements.

16.12.4 Loading

Ore and waste rock will require loading from the open pits into haul trucks. Excavators are selected based on the selective mining capability to minimize loss and dilution in the mill feed, while also achieving sufficiently high mining rates to ensure the lowest possible mine operating unit costs. It is anticipated that four 15 m^3 sized hydraulic shovels, with the capability to excavate in backhoe configurations, will meet these requirements. One wheel loader with a 13.5 m^3 bucket is also specified for waste stripping work. Loading units will also function to re-handle pit material, load overburden and topsoil, pit clean up, crusher feed support, road construction and snow removal. Crusher loading is planned to be done directly via hauler or from the ROM stockpile via the wheel loader.

16.12.5 Hauling

Ore and waste rock are loaded into off-highway rigid frame haul trucks and hauled to specified destinations per the mine production schedule. A haul truck matched to the selected excavators and wheel loader, and with a 140-tonne maximum payload is targeted for this level of mine planning. The size of the fleet is determined by estimating the haulage productivities for ore and waste materials in each period. Haulage productivities are based on simulated hauler cycle times on representative haul routes. These cycle times include loading, hauling, dumping, returning, all wait times and any inefficiencies in the hauling operation.

A secondary fleet of 40 tonne payload articulated haul trucks is also specified to aid in pre-stripping and construction activities within the pit and along the haul roads and the tailings facilities.

16.12.6 Pit Support Services

Pit Support Services include:

- Haul road development and maintenance
- Pit floor and ramp maintenance
- Stockpile maintenance
- Ditching
- Progressive Reclamation
- Mobile Fleet fuel and lube support
- Open pit dewatering
- Topsoil excavation
- Secondary blasting and rock breaking
- Open pit lighting
- Mine safety and rescue
- In pit transportation of personnel and operating supplies
- Snow Removal

Pit support equipment will include track dozers, a backhoe, and a wheel loader for pit floor maintenance, road development and maintenance, and ditching. The road maintenance fleet will also include motor graders and water/gravel trucks.

Ancillary mine equipment will include a small loader and truck fleet, light duty vehicles, utility backhoes, lighting plants, in-pit pumps, and other equipment required to support the mine and maintenance areas of the operation.

16.12.7 Open Pit Dewatering

Preliminary hydrogeological studies (BGC, 2021) indicate that dewatering and depressurization efforts may be required to optimize pit slope angles and improve mining productivity and will be further assessed during the feasibility study.

Pore pressure estimates are based on transient numerical modelling results calibrated to piezometric data collected from 2010 through 2020 and hydraulic conductivity based on packer testing within drillholes from 2010 to 2012.

Groundwater numerical flow modeling was conducted using MODFLOW-SURFACT to predict end of mining pore-pressure conditions and transient groundwater inflow rates to the proposed open pit. Predictive sensitivity analyses indicate that groundwater inflows could range between 0 and 29 L/s as mining progresses below the pre-mining water table surface, with estimates of 3 L/s in the earlier years of the mine plan up to 14 L/s in final years of the mine plan applied for operations planning and costing. Greater groundwater inflow is possible from the north towards to the proposed open pit and is related to the degree of hydraulic connection between Spanish Creek and associated alluvium and fractured bedrock with the proposed open pit.

Predicted pore-pressures for end of mining were applied to the slope stability assessments with up to an additional 50 m of passive depressurization (i.e., non-pumping) within the damage zone due to blasting. Bench designs therefore assume dry conditions and inter-ramp scale designs apply a phreatic surface set-back from the pit slope as simulated in the numerical model. Inter-ramp pit slope angles are based on passively depressurized conditions; locations where active depressurization may provide an opportunity to steepen the design are identified.

To achieve the recommended pit slope angles a pit dewatering program consisting of vertical depressurization wells along the perimeter prior to and during excavation of the pit augmented with horizontal drains in the pit walls during mining. Further studies will be necessary to finalize a pit dewatering plan and evaluate the impacts of the open pit on the regional water balance.

The following areas of uncertainty and risk regarding the estimated hydrogeological conditions are noted.

- Limited understanding of the hydraulic character of alluvium and fractured bedrock within the Spanish Creek valley, including hydraulic properties, heterogeneity, and anisotropy.
- Limited understanding of the hydraulic character of bedrock units, including the influence of faults and fracturing on hydraulic properties, and the heterogeneity and anisotropy of hydraulic properties.

- Limited recent period of record for surface water gauging stations on Spanish Creek.
- Current boundary conditions used to represent Spanish Creek (drain boundary cells) do not permit the simulation of surface water as a source of groundwater recharge.

16.12.8 Mine Maintenance

Mine maintenance activities will be performed in a mine maintenance facility, as well as in the field. The mine maintenance facility is to be located near the mill. Fuel, lube and field maintenance will be performed with a mobile maintenance fleet of equipment.

16.12.9 General Mine Expense and Technical Services

General mine expenses (GME) include the supervision of the direct mining activities, including supervision of the mine fleet maintenance department. GME also includes the technical support requirements from Mine Engineering, Geology and Geotechnical functions.

16.12.10 Mine Buildings

On-site mine service buildings will include a heavy-duty truck shop, mine dry, light duty vehicle shop, wash bay, warehouse/storage facility, fuel depot and distribution, assay laboratory facility, administration-engineering offices, and explosives plant and storage.

16.12.11 Fleet Summary

Primary mining equipment requirements are summarised in Table 16-10.

Table 16-10 Mining Fleet Schedule

FLEET SUMMARY	Y-1	Y1	Y2	Y5	Y8	Y10	Y13
Drilling							
Tracked Rotary Diesel Drill – 200 mm	2	3	3	3	4	3	2
Tracked DTH Diesel Drill – 165 mm	1	1	1	1	1	1	1
Tracked RC Diesel Drill – 144 mm	1	3	3	3	3	3	1
Loading							
Wheel Loader – 13.5 m3 bucket	0	1	1	1	1	1	1
Hydraulic Excavator – 15.0 m3 bucket	2	3	3	3	4	3	1
Hauling							
Rigid Frame Haulers – 136 tonne payload	10	12	12	13	20	20	6
Articulated Haulers – 40 tonne payload	2	2	2	2	2	2	0
Primary Pit Support							
MotorGrader – 4.9 m blade	2	2	2	2	3	3	2
Water / Gravel Truck	2	2	2	2	2	2	1
Track Dozer – 447 kW	1	2	2	2	2	2	1
Track Dozer – 325 kW	1	2	2	2	2	2	2
Wheel Loader – 4.5 m3 bucket	1	2	2	2	2	2	1
Hydraulic Excavator – 3 m3 bucket	1	3	3	3	3	3	1
Fuel / Lube Truck	1	2	2	2	2	2	1
Secondary Pit Support							
Crew Bus	1	2	2	2	2	2	1
Pickup Trucks	8	8	8	8	8	8	4
Light Plants	4	8	8	8	8	8	4
Water Pump	1	1	1	2	2	2	2
Dump Truck	2	2	2	2	2	2	0
Flatbed Picker Truck	1	1	1	1	1	1	1
Emergency Response Vehicle	1	1	1	1	1	1	1
Mine Maintenance Fleet							
Maintenance Trucks	1	2	2	2	2	2	2
Mobile Crane – 35 tonne	1	1	1	1	1	1	1
Float trailer – 150 tonne	0	1	1	1	1	1	1
Forklift Tire Manipulator - 3 tonne	1	1	1	1	1	1	0
Portable Steam Cleaner	1	1	1	1	1	1	0
Scissor Lift	1	1	1	1	1	1	1
Mobile Manlift (4X4)	1	1	1	1	1	1	0
Shovel Float - 300 tonne	0	1	1	1	1	1	1

17 RECOVERY METHODS

17.1 Overall Process Design

Gold occurs as free gold associated with quartz veins and as attachments to and occlusions in pyrite. It recovers well to flotation concentrate and further recovery improvement is seen when regrinding concentrate ahead of cyanidation for improved liberation of fine-grained gold. A primary and scavenger gravity concentration circuit is included for recovery of free gold.

The provided testwork was thoroughly analysed and several options of process routes were assessed in the initial stages of the Prefeasibility 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 project will utilise a capital cost-effective mill design, including a target grind size (P80) of 180 µm, gravity concentration in the primary grinding circuit, 4 stages of flotation, flotation concentrate regrind and leaching, flotation cleaner tailings gravity separation, CIL, carbon elution and gold recovery. Leach-adsorption tailings will be treated for cyanide destruction and sent to the tailings pond.

Key process design criteria are listed below:

- throughput of 20.0 kt/d or 7.3 Mt/a
- crushing availability of 70%
- plant availability of 91.3% for grinding, flotation, leach, adsorption, desorption & cyanide reduction

17.2 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
- gravity and intensive leach circuit
- 4 stages of flotation: rougher flotation, rougher scavenger, and 2 stage cleaner flotation
- flotation concentrate regrind and thickening
- CIL leaching and adsorption
- acid washing of loaded carbon and pressure Zadra elution and electrowinning followed by smelting to produce doré
- carbon regeneration by rotary kiln
- cyanide destruction of tailings using O_2/SO_2 (INCO process)
- thickening and cleaner tailings gravity concentration
- tailings storage facility

17.2.1 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	t/d	20,000
Gold Head Grade – Design	g/t Au	0.92
Silver Head Grade – Design	g/t Au	0.72
Crushing Plant Availability	%	70
Mill Availability	%	91.3
Bond Crusher Work Index (CWi)	kWh/t	17.9
Bond Rod Work Index (RWi)	kWh/t	14.7
Bond Ball Mill Work Index (BWi)	kWh/t	13.8
Bond Abrasion Index (Ai)	g	0.21
JK Dropweight Parameter Axb		39.2
Primary Crusher Type		gyratory
Material Specific Gravity	t/m^3	2.68
SAG Mill Dimensions		30 ft dia. X 22 ft EGL
SAG Mill Installed Power	MW	10.5, with VSD
SAG Mill Discharge Density	% w/w	70
SAG Mill Ball Charge	% v/v	12
Ball Mill Dimensions		20 ft dia. X 26.5 ft EGL
Ball Mill Installed Power	MW	5.25
Ball Mill Ball Charge	% v/v	29
Circulating Load	%	350
Classification Cyclone Overflow Density	% w/w	35
Primary Grind size (P80)	µm	180
Gravity Concentrators	#	4
Gravity Circuit Gold Recovery	%	18
Gravity Circuit Silver Recovery	%	6
Rougher/Scavenger Flotation Cell Type		DFR
Rougher/Scavenger Flotation Residence Time	min	18
Cleaner Flotation Cell Type		DFR
Cleaner Flotation Residence Time	min	13
Re-Cleaner Flotation Cell Type		DFR
Re-Cleaner Flotation residence Time	min	13
Final Flotation Concentrate Mass Recovery	%	3

Design Parameter	**Units**	**Value**
Cleaner Tailings Thickener Underflow Density	% w/w	45
Scavenger Gravity Concentrators	#	2
Scavenger Gravity Circuit Mass Pull	%	5
Concentrate Regrind Mill Type		HIGmill
Concentrate Regrind Mill Quantity	#	1
Concentrate Regrind Installed Power, each	MW	0.5
Concentrate Regrind Product Size (P80)	µm	22
Pre-Leach Thickener Underflow Density	% w/w	50
Flotation Concentrate Leach Pre-Aeration Tank	#	1
Flotation Concentrate Leach Pre-Aeration Tank	h	12
Flotation Concentrate CIL Tanks	#	8
Flotation Concentrate Leach Residence Time	h	48
Elution Carbon Batch Size	t	5
Elution Strips Per Week		7
Detox Residence Time	min	120
Detox WAD Cyanide Feed to Circuit	mg/L CN_{WAD}	500
Detox WAD Cyanide Discharge Target	mg/L CN_{WAD}	<2.0

Reagents in use in the process and the consumption rates are shown in Table 17-2.

Table 17-2 Reagent and Grinding Media Consumption

Reagent Description	Consumption g/t of mill feed
Collector (PAX)	135
Frother (MIBC)	71
Depressant (CMC)	65
Quick lime	112
NaCN	117
Oxygen	82
Activated carbon	1
NaOH	87
HCl	16
SMBS	162
$CuSO_4$	7
SAG mill grinding media	332
Ball mill grinding media	278
Regrind mill media	2

17.2.2 Primary Crushing & Stockpiling

The crushing circuit is designed for an annual operating time of 6,132 h/a or 70% availability.

Material is hauled from the open pit and fed to the primary crusher at 1,190 t/h. The crushed material is conveyed to a covered stockpile that provides approximately 12 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 is equipped with apron feeders to regulate feed at 913 t/h into the SAG mill. Crushed material is drawn from the stockpile by two apron feeders and feeds the mill circuit via the SAG mill feed conveyor.

The materials handling and crushing circuit includes the following key equipment:

- primary gyratory crusher
- mill feed apron feeders (equipped with VSDs)
- materials handling equipment

17.2.3 Grinding Circuit

The grinding circuit consists of a SAG mill followed by a ball mill in closed circuit with hydrocyclones. The circuit is sized based on a SAG F80 of 118 mm and a ball mill product P80 of 180 µm. The SAG mill slurry discharges through a trommel where the pebbles are screened and recycled to a pebble crusher before returning to the SAG mill via conveyor. Trommel undersize discharges into the cyclone feed pumpbox.

The ball mill is fed by the cyclone underflow and discharges through a trommel. The oversize is screened out and discharged to a scats bunker. Trommel undersize discharges into the cyclone feed pumpbox.

Water is added to the cyclone feed pumpbox to obtain the appropriate density prior to pumping to the cyclones and gravity concentrators. Cyclone overflow gravitates to the rougher flotation circuit via a trash screen.

The grinding circuit includes the following key equipment:

- 10.5 MW SAG mill (equipped with VSD)
- 600 kW Pebble crusher
- 5.25 MW ball mill
- classification cyclone cluster
- trash screen

17.2.4 Gravity and Intensive Leach Circuit

The gravity circuit treats 32% of the mill circulating load and the gravity screen is fed from the cyclone feed pumpbox. Screen oversize material returns to the cyclone feed pumpbox. Undersize material reports to 4 gravity concentrators which feed the intensive leaching circuit. The intensive leach solution is treated through an electrowinning cell for recovery of gold and silver. The gravity electrowinning cell is located within the gold room.

The gravity and intensive leach circuit includes the following key equipment:

- gravity concentrators (4)
- Intensive leach reactor
- Gravity eluate tank
- Electrowinning cell

17.2.5 Flotation Circuit

The flotation circuit comprises of 4 stages: rougher flotation, rougher scavenger flotation, cleaner flotation, and recleaner flotation. The equipment selected for all flotation stages is based on DFRs (Direct Flotation Reactors) from Woodgrove Technologies.

Ore in the overflow stream from the classification cyclones feeds three DFR rougher flotation cells. The rougher flotation circuit requires an addition of 90 g/t of PAX and 50 g/t of MIBC. Concentrate is collected along the bank and reports the cleaner circuit. Rougher tailings from the third cell report to the scavenger circuit.

The scavenger circuit includes five DFR cells. Concentrate from each cell is collected and sent to the cleaner flotation circuit. Tailings from the fifth cell in the bank report to the tailings pond.

The cleaner flotation circuit includes nine DFR cells with reagent additions of 35 g/t PAX, 15 g/t MIBC, and 50 g/t CMC. Concentrate from each of the cells is collected and pumped to the recleaner flotation circuit. Tailings from the ninth cell in the bank report to the cleaner and recleaner tailings thickener.

The recleaner flotation circuit includes five DFR cells with reagent additions of 10 g/t PAX, 6 g/t MIBC, and 15 g/t CMC. Concentrate from each cell is collected and sent to the cleaner flotation concentrate cyclone cluster and regrind mill before reporting to the flotation concentrate thickener and leach circuit. Tailings from the fifth cell in the bank report to the cleaner and recleaner tailings thickener.

The cleaner and recleaner tailings thickener has an underflow density of 45% solids. The underflow reports to the scavenger gravity concentration circuit.

The scavenger gravity concentration circuit has 2 gravity concentrators and a mass pull of 5%. Gravity circuit concentrate reports to the regrind circuit before leaching and the tailings reports to the tailings pump box, which also receives detoxified tailings.

The flotation circuit includes the following key equipment:

- Rougher DFR feed head tank
- Rougher DFRs (3)
- Scavenger DFRs (5)
- Cleaner DFR head tank
- Cleaner DFRs (9)
- Recleaner flotation head tank
- Recleaner DFRs (5)
- Cleaner and recleaner tailings thickener
- Scavenger gravity concentrators (2)

Selection of DFRs for flotation is considered a low technology risk, as there are sufficient installations worldwide for this to be considered proven technology. Pilot testing is planned of the feasibility study phase with the objective of optimising sizing of the cleaner circuit and downstream concentrate processing equipment.

17.2.6 Cleaner Flotation Regrind and Dewatering Circuit

Cleaner flotation concentrate reports to the cyclone cluster. The cyclone cluster underflow reports to the regrind mill (HIGmill) configured in an open circuit. Regrind mill product at a P80 grind size of 22 µm is combined with cyclone overflow and reports to the concentrate thickener. Thickener underflow at 50% solids reports to the leach circuit.

The cleaner regrind and dewatering circuit includes the following key equipment:

- Cyclone cluster
- Regrind mill
- Concentrate thickener

17.2.7 Flotation Concentrate Leach & Adsorption Circuit

The flotation concentrate leach-adsorption circuit consists of one pre-aeration tank and eight CIL tanks. The circuit is fed by the flotation concentrate regrind and dewatering circuit. The CIL tanks are 265 m^3 each, with a total circuit residence time of 48 hours at 50% w/w solids.

Oxygen is sparged into the pre-aeration and initial leach tanks to maintain adequate dissolved oxygen levels for leaching at 20 ppm. Hydrated lime is added to the pre-aeration tank and first three CIL tanks to further refine the operating pH at a rate of 1.1 kg/t CIL feed. Cyanide solution is added to the first three leach tanks at 3 kg/t CIL feed. Fresh/regenerated carbon from the carbon regeneration circuit is returned to the last tank of the CIL circuit and is advanced counter-currently to the slurry flow by pumping slurry and carbon. Slurry from the last CIL tank is pumped to the cyanide detox tank.

The intertank screen in each CIL tank retains the carbon whilst allowing the slurry to flow by gravity to the downstream tank. This counter-current process is repeated until the loaded carbon reaches the first CIL tank. Recessed impeller pumps are used to transfer slurry between the CIL tanks and from the lead tank to the loaded carbon screen mounted above the acid wash column in the elution circuit.

The flotation concentrate leach and carbon adsorption circuit includes the following key equipment:

- CIL tanks and agitators
- loaded carbon screen
- intertank carbon screens
- carbon sizing screen

17.2.8 Cyanide Detoxification

Leach-adsorption tailings feed the detoxification circuit at 45% w/w solids.

The detoxification circuit consists of one tank with a residence time of approximately 120 mins to reduce weak acid dissociable cyanide (CN_{WAD}) concentration from 500 mg/L to less than 2 mg/L to comply with environmental requirements prior to deposition in the tailings pond.

Cyanide destruction is undertaken using the SO_2/O_2 method. The reagents required are oxygen, lime, copper sulphate, and sodium metabisulphite (SMBS). The cyanide destruction tank is 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 tank, the tailings pass through the carbon safety screen and collect in the tailings pumpbox before being pumped to the tailing pond.

The main equipment in this area includes:

- cyanide destruction tank and agitator
- carbon safety screen

17.2.9 Carbon Acid Wash, Elution & Regeneration Circuit

17.2.9.1 Carbon Acid Wash

Prior to the gold stripping stage, loaded carbon is 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 flows by gravity to the acid wash column. Entrained water is drained from the column and the column is refilled from the bottom up with the hydrochloric acid solution. Once the column is filled with the acid, it is left to soak, after which the spent acid is rinsed from the carbon and discarded to the cyanide destruction tank.

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

The main equipment in this area includes:

- acid wash carbon column

17.2.9.2 Gold Stripping (Elution)

The gold stripping (elution) circuit uses a pressure Zadra process.

Strip solution (eluate) will be made up in the strip-solution tank using raw water dosed with a dilute sodium hydroxide and cyanide solution to form an electrolyte for the electrowinning process. This solution will be circulated through the elution column via an eluate heater, which heats the solution, the carbon, and the column to 130°C. The elution system will be pressurized to keep the solution from flashing to steam in the heater or elution column.

A recovery heat exchanger will transfer heat from the hot pregnant solution exiting the column to the incoming solution before passing through the solution heater. This will reduce the energy required to maintain the solution temperature and cool the pregnant solution before it enters the electrowinning cell. Once the required system temperature is reached, the hot pregnant eluate solution will be directed to the electrowinning cell, where the metals will be plated onto cathodes. Solution continues to circulate through the elution column and electrowinning cell. The process will continue to deposit metals into the electrowinning cell for a maximum of 16 hours.

Upon completion of the cool down sequence, the carbon is hydraulically transferred to the carbon regeneration kiln feed hopper via a de-watering screen.

The stripping circuit includes the following key equipment:

- elution column
- strip solution heater (propane-fired) with heat exchanger
- strip solution tank

17.2.9.3 Carbon Reactivation

Carbon is reactivated in a rotary kiln. Dewatered barren carbon from the stripping circuit is held in a feed hopper. A screw feeder metres the carbon into the reactivation kiln, where it is heated to 750°C in an atmosphere of superheated steam to restore the activity of the carbon.

Carbon discharging from the kiln is quenched in water and screened on a carbon sizing screen to remove undersized carbon fragments. The undersize fine carbon gravitates to the carbon safety screen, whilst carbon screen oversize is directed to the CIL circuit.

As carbon is lost by attrition, new carbon is added to the circuit using the carbon quench tank. The new carbon is then transferred along with the regenerated carbon to feed the carbon sizing screen.

The carbon reactivation circuit includes the following key equipment:

- carbon dewatering screen
- regeneration kiln (propane-fired) including feed hopper and screw feeder
- carbon quench tank

17.2.10 Electrowinning & Smelting

Gold and silver are recovered from the pregnant solution by electrowinning and smelted to produce doré bars. Gold is deposited on the electrowinning cell cathodes and the resulting barren solution is returned to the leach circuit.

The gold-rich sludge is washed off the steel cathodes in the electrowinning cells using high-pressure spray water and gravitates to the sludge hopper. Sludge is dewatered in a filter press and then transported manually using a tray to the drying oven

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 moulds on a mould 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 cells, drying oven, and smelting-furnace off gasses.

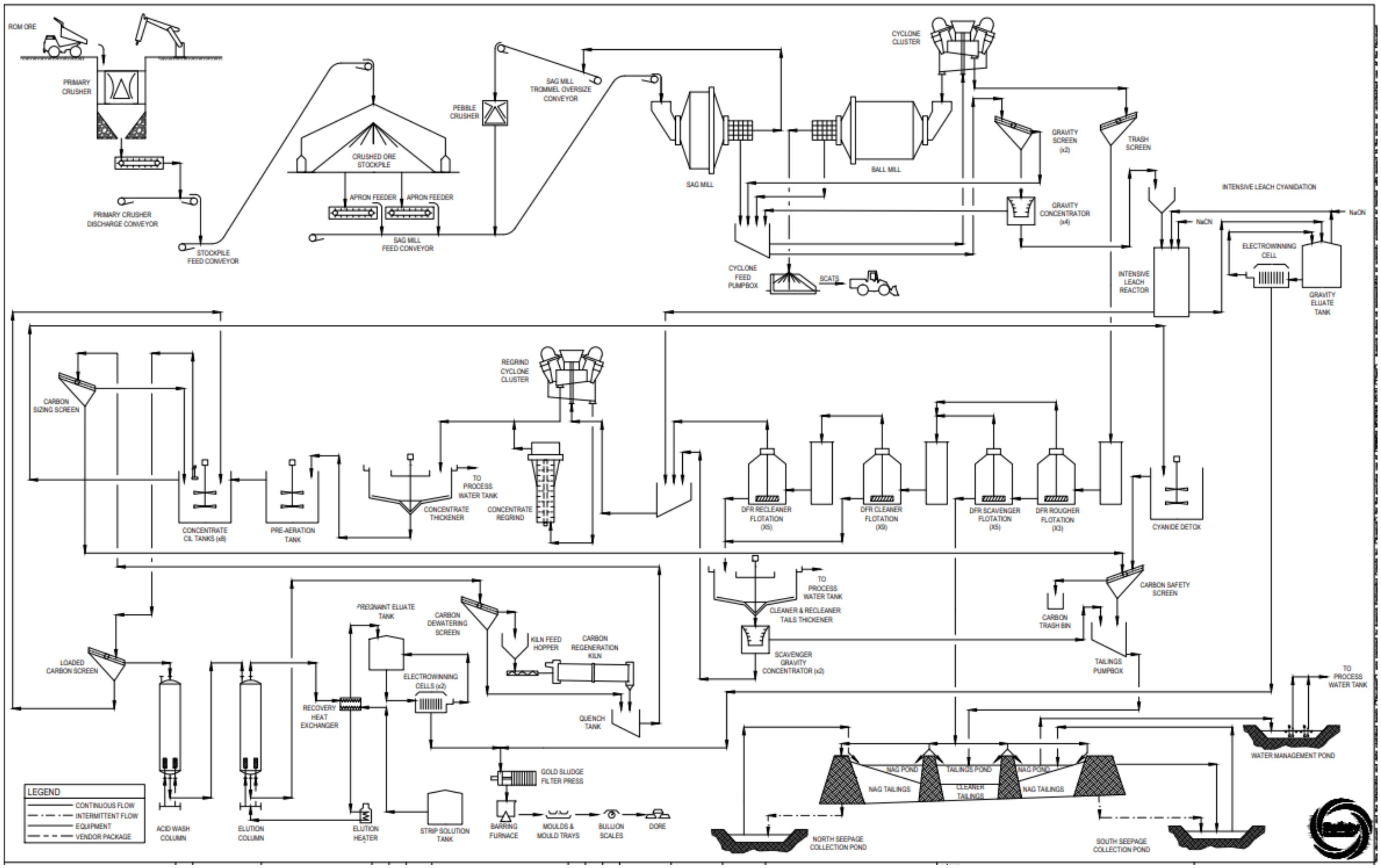
The electrowinning and smelting process takes place within a secure and supervised gold room.

The electrowinning circuit and gold room include the following key equipment:

- electrowinning cells with rectifiers (2)
- sludge pressure filter
- flux mixer
- barring furnace with bullion moulds and slag handling system
- bullion vault and safe
- dust and fume collection
- gold room security system

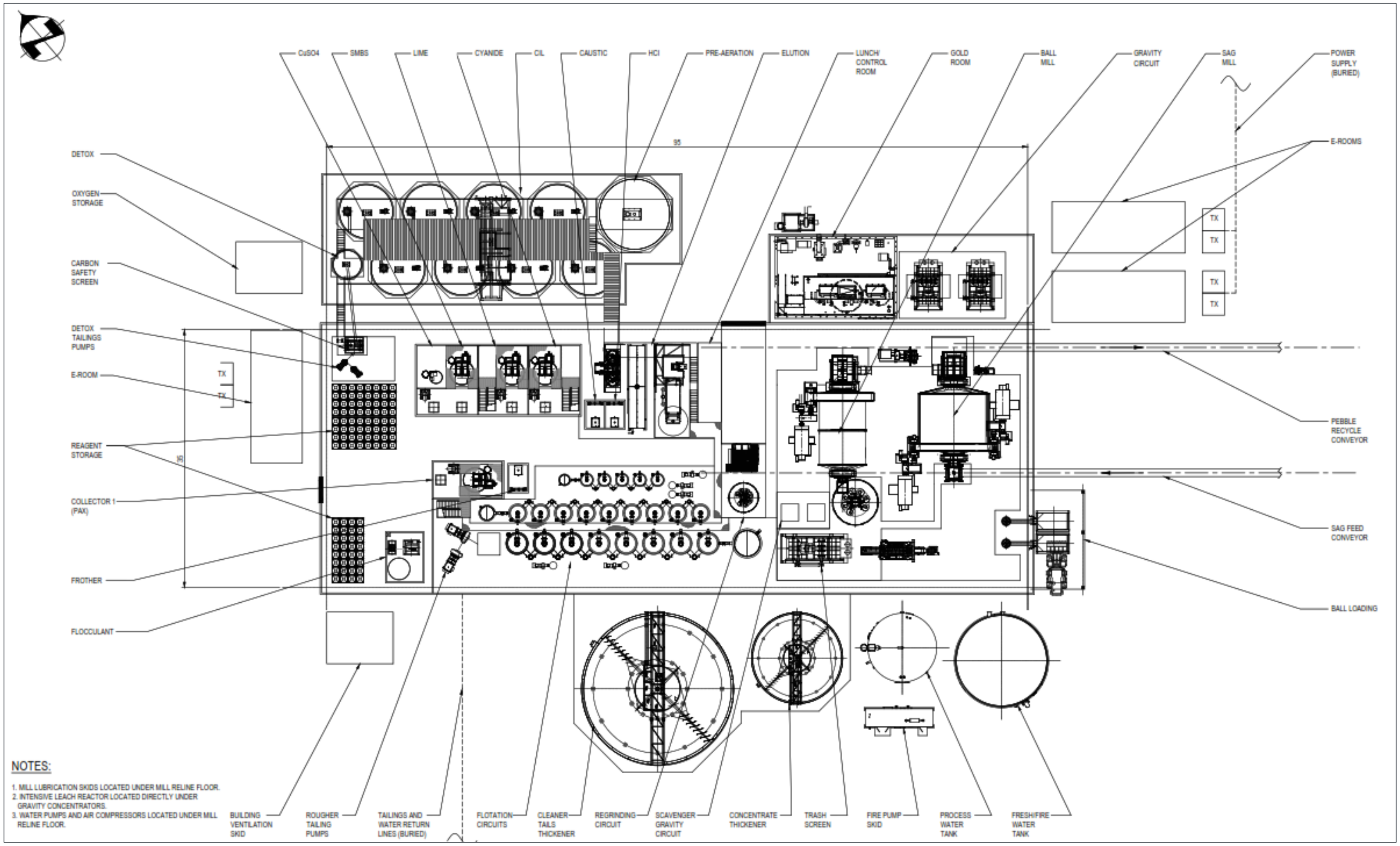
17.2.11 Flowsheet & Layout Drawings

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.



Source: Ausenco, 2020.

Figure 17-1 Overall Process Flow Diagram



Source: Ausenco, 2020.

Figure 17-2 Overall Plant Layout

17.3 Reagent Handling & Storage

Each set of compatible reagent mixing, and storage systems are located within containment areas to prevent incompatible reagents from mixing and to contain any spillage. Storage tanks are 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 are located throughout the facilities. Sumps and sump pumps are provided for spillage control.

17.3.1 Lime

Hydrated lime powder is delivered in bulk bags which are lifted using a frame and hoist and emptied into the lime silo. Hydrated lime is extracted from the lime silo and fed by a screw feeder into the lime mix and storage tank. The lime distribution pumps move lime slurry from the lime mix and storage tank to the CIL circuit and cyanide detoxification.

17.3.2 Sodium Cyanide (NaCN)

Sodium cyanide is delivered to site as briquettes in bulk bags, which are lifted using a frame and hoist. Periodically a single bag is placed on the sodium cyanide bag breaker on top of the tank. The solid reagent falls into the tank and is dissolved with process water while being agitated to achieve the required dosing concentration.

After the mixing period is complete, cyanide solution is transferred to the cyanide storage tank using a transfer pump. Sodium cyanide is delivered to the CIL circuit, strip solution tank, and intensive leach reactor.

17.3.3 Sodium Hydroxide (NaOH)

Sodium hydroxide (caustic soda) is delivered in totes as solution. Dosing pumps automatically deliver the reagent to the required locations—elution circuit, eluate tanks and ILR —to ensure the dosing requirements are met.

17.3.4 Hydrochloric Acid (HCl)

Hydrochloric acid is delivered in totes as solution. Hydrochloric acid is delivered to the acid wash column using the hydrochloric acid dosing pump.

17.3.5 Copper Sulphate Pentahydrate

Copper sulphate pentahydrate is delivered in solid crystal form in 25kg bags. Process water is added to the agitated copper sulphate storage tank. Bags are manually lifted and placed on the copper sulphate bag breaker on top of the tank. The solid reagent falls into the tank and is dissolved in water to achieve the required dosing concentration.

Copper sulphate is delivered to the cyanide detoxification circuit using the copper sulphate dosing pump. An extraction fan is provided over the copper sulphate bag breaker/mixing tank to remove reagent dust that may be generated during reagent addition/mixing.

17.3.6 Sodium Metabisulphite (SMBS)

SMBS is delivered in the form of solid flakes in bulk bags. Process water is added to the agitated SMBS mixing tank. Bags are lifted using a frame and hoist into the SMBS bag breaker on top of the tank. The solid reagent falls into the tank and is dissolved in water to achieve the required concentration. After the mixing period is complete, SMBS is delivered to the cyanide detoxification circuit using the SMBS dosing pump. An extraction fan is provided over the SMBS mixing tank to remove SO2 gas that may be generated during mixing.

17.3.7 Activated Carbon

Activated carbon is delivered in solid granular form in bulk bags. When required, the fresh carbon is introduced to the carbon quench tank, or directly to the final CIL tank.

17.3.8 Flocculant

Powdered flocculant is delivered to site in 25kg bags and stored in the plant building. A self-contained mixing and dosing system are installed, 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 is loaded into the flocculant storage hopper. Dry flocculant is pneumatically transferred into the wetting head, where it is contacted with water.

Flocculant solution, at 0.50% w/v, is agitated in the flocculant mixing tank for a pre-set period. After a pre-set time, the flocculant is transferred to the flocculant storage tank using the flocculant transfer pumps. Flocculant is further diluted just prior to the addition point. A common system supplies flocculant to the cleaner and recleaner tailings thickener and the CIL feed thickener.

17.3.9 Depressant (CMC)

Powdered CMC is delivered to site in bulk bags and stored in the plant building. A self-contained mixing and dosing system are installed, including a CMC storage hopper, CMC screw feeder, CMC preparation water heater, CMC mixing tank, CMC transfer pump, CMC storage tank and CMC dosing pump. Powdered CMC is loaded into the CMC storage hopper using the CMC hoist. Dry CMC is pneumatically transferred into the wetting head, where it is contacted with water.

CMC solution, at a dosing strength of 4% w/w, is agitated in the CMC mixing tank for a pre-set period. After a pre-set time, the CMC is transferred to the CMC storage tank using the CMC transfer pumps. The CMC dosing pump supplies CMC to the flotation area.

17.3.10 Collector 1 (PAX)

PAX is delivered in granular powder form in bulk bags. Process water is added to the agitated PAX storage tank. Bags are lifted using a frame and hoist into the PAX bag breaker on top of the tank. The solid reagent falls into the tank and is dissolved in water to achieve the required dosing concentration.

The mixing tank is ventilated using the PAX tank fan to remove any carbon disulphide gas. PAX is delivered to the flotation circuit using the PAX dosing pump.

17.3.11 Frother (MIBC)

MIBC is delivered as a liquid in totes. It is used as received and without dilution. Diaphragm-style dosing pumps deliver the reagent to the required locations within the flotation circuit.

17.3.12 Oxygen

Oxygen is injected into the pre-aeration and flotation concentrate leach tanks to achieve a dissolved oxygen level of >20 ppm. For this purpose, liquid oxygen is supplied by the vendor in a pressurised tank. From there, it goes through vaporisers, then feeds the leach and detoxification tanks as required.

17.3.13 Gold Room Smelting Fluxes

Borax, silica sand, sodium nitrate, and sodium carbonate are delivered as solid crystals/pellets in bags or plastic containers and stored in the gold room until required.

17.4 Services & Utilities

17.4.1 Process/Instrument Air

High-pressure air at 750 kPag is produced by compressors to meet plant requirements. The high-pressure air supply is dried and used to satisfy both plant air and instrument air demand. Dried air is distributed via the air receivers located throughout the plant.

17.4.2 Low Pressure Air

Low-pressure air at 300 kPag for flotation is supplied by air blowers to the rougher and cleaner flotation circuits.

17.5 Water Supply

17.5.1 Fresh Water Supply System

Fresh water is supplied from two freshwater wells to a fresh/ fire water tank. Fresh water is 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)

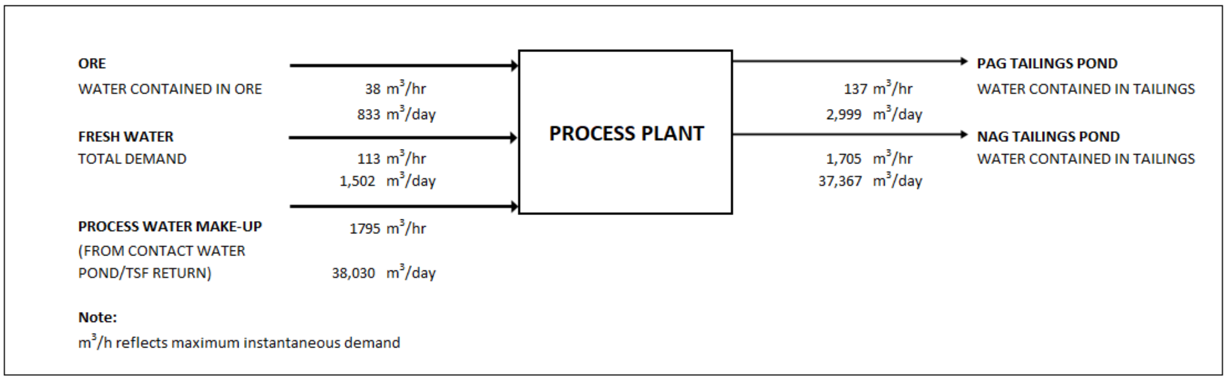
Potable water is pumped from the fresh fire water tank to a sterilization skid and stored in a potable water tank before being sent to potable water uses via the potable water pump.

17.5.2 Process Water Supply System

Overflow from the thickeners, as well as water stored in the water management pond meet the majority of the process water requirements. Raw water and contact water provide any additional make-up water requirements. The water management pond is fed with clarified water from the non-acid generating sections of the tailings pond. Seepage water from the north and south seepage ponds is pumped back into the non-acid generating sections of the tailings pond.

17.5.3 Plant water Demand

The overall projected plant water balance is shown in Figure 17-3.



Note: Figure prepared by Ausenco, 2020

Figure 17-3 Plant Water Balance

17.5.4 Projected Energy Requirements

17.5.4.1 Electrical

The installed power for the process plant is estimated to be 26.2MW, and power consumption is $1.73/t of material treated. The project electrical power cost is 0.065 $/kWh.

The installed power for the G&A areas is 1.47MW and power consumption is $0.07/t treated.

17.5.4.2 Propane

Peak propane demand is estimated at 13.5 m^3 per day. The elution heaters and carbon regeneration kilns account for approximately 5 m^3 per day of the propane demand. The balance is for the process plant and infrastructure HVAC systems, which operate seasonally.

18 PROJECT INFRASTRUCTURE

18.1 Introduction

This Section discusses the Project infrastructure, including on-site infrastructure, the tailing storage facility (TSF), water treatment plant (WTP), and water management systems and process and non-process related site infrastructure. During operations SMG will transport mine operational staff from local pick-up hubs to the Project site. A temporary 265-man camp will be provided during construction and an existing 50-man permanent camp will be available at the site during construction and for the operational phase of the project.

On-site Infrastructure to support the Spanish Lake Project will comprise site civil work, site roads, site facilities and buildings, a water management system, TSF, WTP, and site electrical facilities and distribution. The location of these facilities is depicted in Figure 18-1.

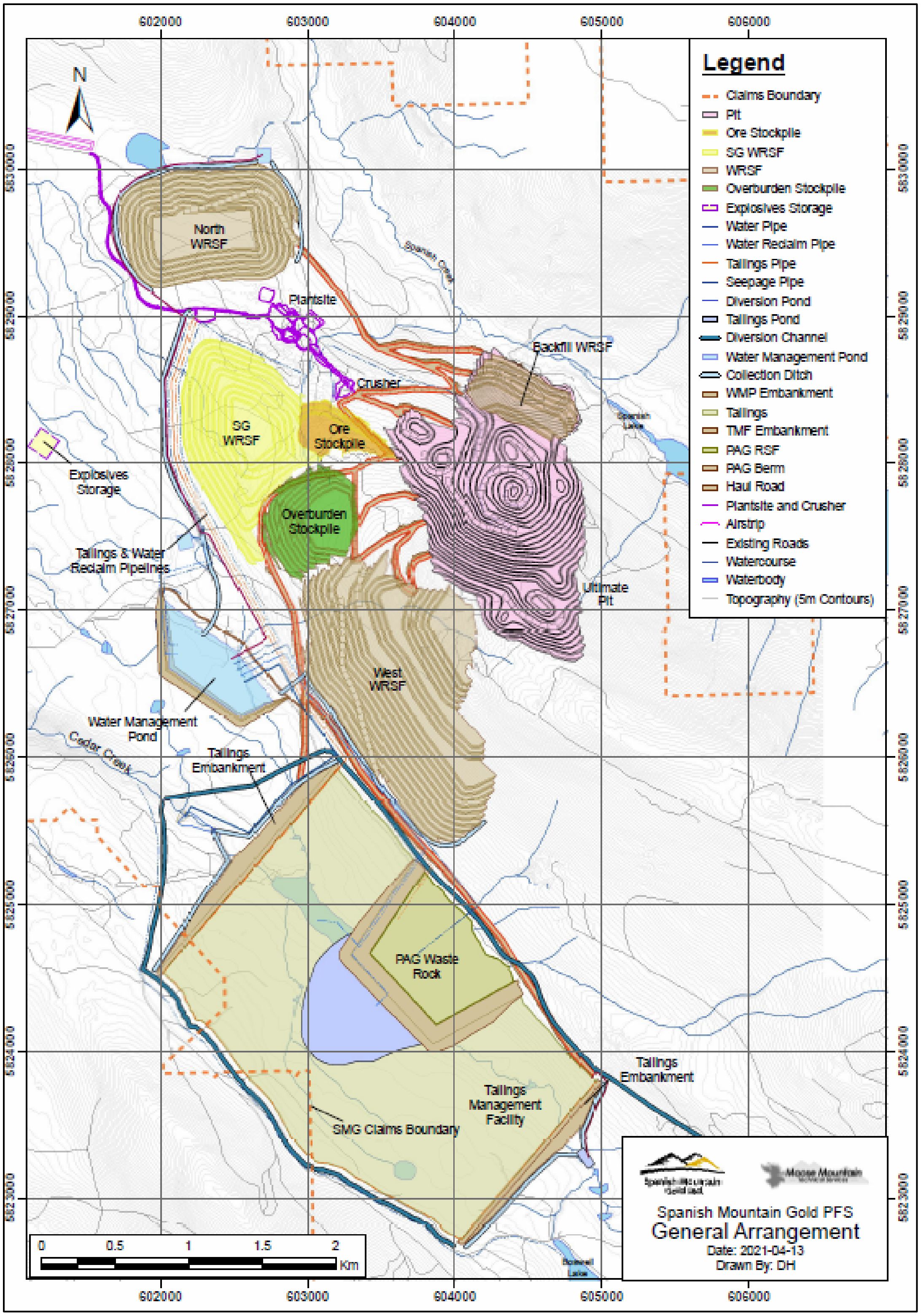


Figure 18-1 Site General Arrangement

Off-site infrastructure includes construction of a new 138 kV Substation near Highway 97 (by BC Hydro) and a new 138kV transmission line will be installed between the new 138kV Substation and the receiving substation at the Spanish Mountain mine site.

The Project will incorporate features common to mines in British Columbia, including:

- Storage facilities for consumables will be sized for short-term supply requirements (e.g., mobile diesel fuel tanks, mobile propane tanks and warehouse) as these can be replenished on a regular basis;
- Propane will be used for heating buildings;
- Major buildings, including the process plant, and plant site ancillary buildings will not be connected with heated, enclosed corridors. Instead, buildings will be located close to one another, limiting travel distance for personnel to access most facilities;
- Wherever practical, equipment and buildings will be pre-assembled off site and delivered to site in modules or skids to minimize site erection time;
- Facility layout will accommodate snow clearing and mitigation of drifting effects, such as simple building designs;
- Most fixed equipment will be housed in heated or unheated structures or shelters for protection from wind-chill effects, and to enable personnel access and operation in extreme weather conditions;
- Open areas for laydown will have a granular pad over native ground.

Site civil work includes designs for the following infrastructure:

- Light vehicle and heavy equipment roads;
- Growth media stripping and stockpiling;
- Site drainage and water collection around the plant site;
- Site facilities;
- High voltage substation platform.

Site facilities will include both infrastructure facilities and process facilities:

- The infrastructure facilities will include the administration offices, truck shop and warehouse, mine dry, fuel storage and distribution, permanent camp facility, and cold storage warehouse;
- The process facilities will include the process plant, crusher facility, process plant workshop and assay laboratory;
- Common support services such as potable water, firewater, and sewage treatment will be provided by stand-alone equipment and systems. Waste generated from operations will be managed on site. Depending on category, wastes will be incinerated, or shipped off site for proper disposal at approved facilities.

18.2 Site Facilities

Figure 18-2 presents the plant site general arrangement.

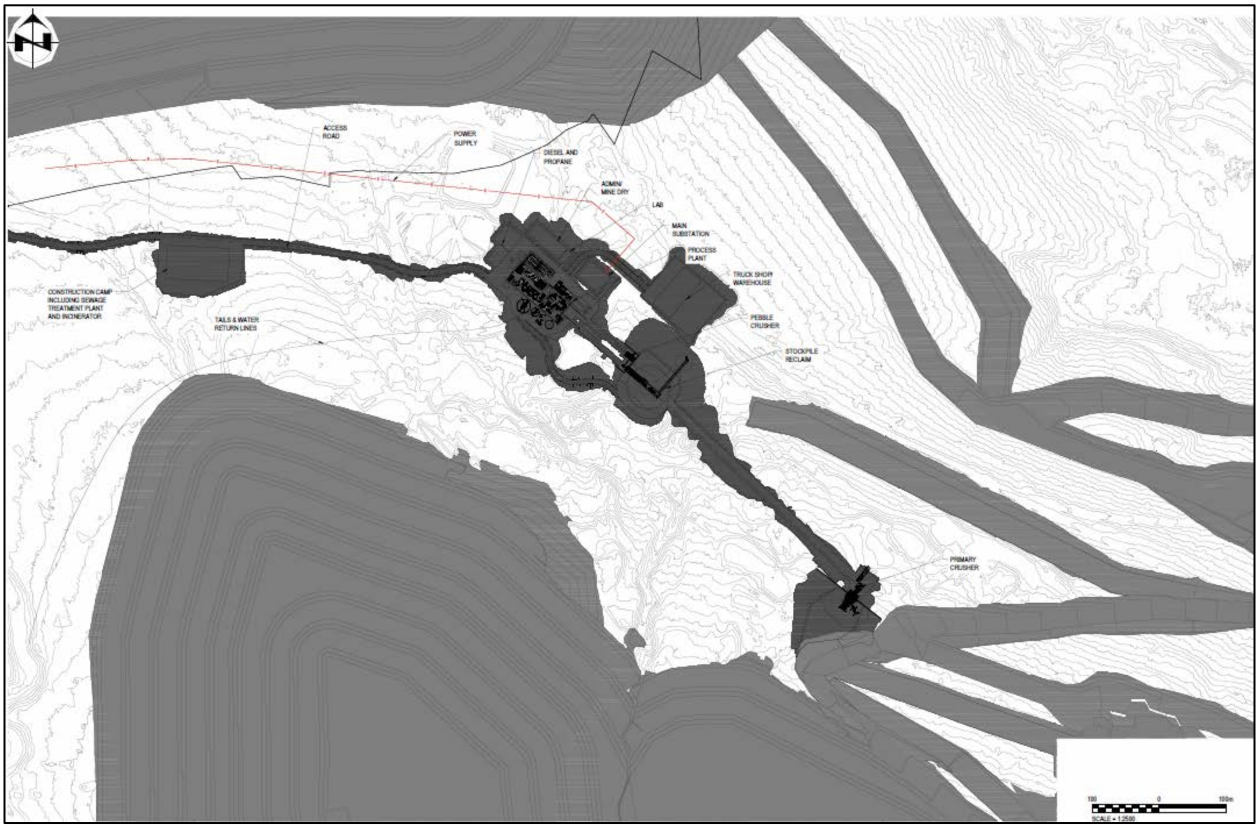


Figure 18-2 Plant Site General Layout

The following infrastructure facilities were included:

- Truck shop and warehouse and including heavy vehicle repair bay, tire bay, wash bay, lubrication bay, light vehicle repair bay and maintenance shops;
- Administration/mine dry building;
- Assay laboratory;
- Cold storage warehouse;
- Access road and site roads;
- Fuel storage;
- Accommodations;
- Process Plant;
- Solid waste disposal
- Raw water supply;
- Security/gatehouse;
- Site fencing;
- Construction laydown yard;
- Sewage collection and treatment.

The general layout of the plant site was based on several criteria:

- Compact footprint for minimal land disturbance and maximum site operations efficiency;
- Compact building sizes and layout for maximum energy efficiency;
- Efficient facility access for personnel and vehicles during construction and operations;
- Minimal visual and noise impact from operations on personnel accommodations.

18.2.1 Plant Site

The Spanish Mountain Plant Site is located approximately 1.0km west of the pit and has been positioned to avoid areas identified to have artesian groundwater conditions as well as previously excavated areas now functioning as manmade ponds. The plant site uses a stepped layout which positions infrastructure pads to better balance cut and fill quantities.

18.2.2 Truck Shop and Warehouse

The truck shop will be a pre-engineered building 72 m long x 34 m wide and fitted with insulated metal cladding and two 30 t overhead cranes. The building was sized to accommodate 140 t haul trucks and it will have service bays for lubrication, truck washing, heavy and light vehicle repair, welding, and tire replacement. The facility will also have space for maintenance workshops and a warehouse facility.

18.2.3 Administration/Dry Building

The administration and dry building will be a modular building supported on concrete spread footings, complete with furniture and equipment. The building footprint will be 1,008 m^2. The administration building includes working space and offices for engineering, technical, surveying, and administration personnel. The offices of the general manager, mine manager, mill superintendent, mine operations superintendent, maintenance superintendent, and mine supervisors will be in this building. A first aid safety area, control station, kitchen, and lunchroom facilities are also in the building.

18.2.4 Assay Laboratory

The assay laboratory will be a modular building, 160 m^2 in area, and will be situated adjacent to the administration building, close to the process building. The building will house all laboratory equipment for the daily operational process control including the metallurgical and geological requirements.

18.2.5 Cold Storage Warehouse

The cold storage warehouse will be a pre-engineered light steel framed structure with an un-insulated fabric cover. The building will have a footprint of 800 m^2 and will be supported on pre-cast concrete lock blocks on a prepared gravel surface.

18.2.6 Access Road, Site Roads, and Spanish Lake Road Re-routing

The access road upgrade and extension from Spanish Mountain 13000 Forest Service Road to the plant site, and the rerouted Spanish Lake Road will be constructed as soon as possible to maintain public access around the site.

The current plan is for the access road civil contract to be tendered separately to the civil works on-site. Battery limit of works will be the entrance to Spanish Lake Road to the plant site as depicted on the site layout drawing. Ausenco will manage the access road and rerouting construction execution.

The Spanish Mountain Property can be accessed along a 1.9 km Main Access Road, which ties into the existing Spanish Lake Road west of Hepburn Lake. The main access road provides access to the process plant site, accommodations, and laydown yard after passing the gatehouse for the property.

The site roads provide access to the on-site facilities for the appropriate vehicles associated with facility usage. These roads will also provide access to the facilities being constructed.

Two categories of roads are proposed within the site: in-plant roads and maintenance roads. In-plant roads are two lane roads consisting of 3.0m lanes and 0.5m shoulders. The maintenance roads are single lane roads consisting of a 4.0m lane and 0.5m shoulders.

Immediately after the tie-in point to the main access road, Spanish Lake Road has been re-routed to avoid Project Infrastructure such as Rock Storage Facilities (RSF) and the ultimate pit. The re-routing covers a 5.4 km portion of Spanish Lake Road, shifting the road north around the project site and ties the road back into the existing portion south of Spanish Lake.

18.2.7 Fuel Storage

Diesel fuel, the primary Project fuel, will be supplied in mobile 75,000 litre tanks and stored in a concrete bunded fuel storage area located at the plant site. The anticipated mining fleet diesel consumption was calculated to be 16 Ml per year for the first six years and then it will ramp-up to 27 Ml from Year 7 onwards. The fuel storage area will initially store four mobile fuel storage tanks and expanded later to store seven as fuel consumption ramps-up. The mobile fuel tanks will be double-walled and will be equipped with prepackaged fuel unloading modules. These tanks will also supply fuel for the plant site mobile equipment. Fuel will be trucked to storage tanks for other facilities such as emergency generators and incinerators.

Liquid propane (LPG) for building heating will be provided by contractor road tanker, from supply depots in Williams Lake.

18.2.8 Accommodations

A construction camp capable of accommodating and catering for 265 persons will be assembled from prefabricated modules. Catering will also be provided for the 50-man camp. There will be a core complex with dining, kitchen, and limited recreational facilities. During operations the 50-man camp will be operational, the majority of operational staff will be transported to site from local hubs. If additional accommodation is required rooms in the temporary construction camp can be utilized.

18.2.9 Primary Crusher, Stockpile and Process Plant

The primary crusher area will feature a concrete tower to support the crusher and provide a dump pocket above the crusher and a crushed ore pocket beneath.

The process plant building will have an approximate area of 3,325 m^2, and will house all the milling, flotation, and reagent equipment. The building will be divided into two sections. The first section will contain the mills and will have dimensions of 45 x 35 m; the second section will contain the flotation, regrind and reagent equipment and will be 50 x 35 m in size. Both sections will be serviced by overhead cranes. This building will be a pre-engineered steel frame and metal-clad building with internal insulation to reduce heat loss. Buildings housing the ILR circuit (234 m^2) and gold room (216 m^2) will be attached to the main process building. The buildings will be heated using unit heater and air handling systems utilizing propane as a fuel.

18.2.10 Solid Waste Disposal

An incinerator will be used for the incineration of non-hazardous, combustible waste materials and will be located within the accommodation complex.

No provision has been made for an on-site landfill. Inert solid waste will be collected and transported off-site to the nearest landfill. Hazardous waste will be collected and transported off-site for disposal.

18.2.11 Fire, Raw Water and Potable Water

Water for fire protection will be stored in the freshwater tank located at the process plant and from there it will be distributed on demand.

Fresh water will be supplied from well pumps and will be sterilized before being stored in a potable water tank and distributed to the process plant, plant site buildings, and camp.

18.2.12 Security Gatehouse and Site Fencing

A small modular security gatehouse (42 m^2) located at the entrance of the mine site will be provided. The site will be fenced with approximately 4 km of chain mesh fencing 1.8 m high.

18.2.13 Laydown Area

A laydown area will be provided adjacent to the plant site access road to receive, organise, and store construction materials during pre-production and for any overflow materials during operations.

18.2.14 Wastewater Treatment Systems

Wastewater or sewage generated on-site will be treated at a sewage treatment facility. Treated effluent generated at the sewage treatment plant will be compliant with applicable regulations.

All potable water that is generated and consumed on-site for domestic use is expected to report to the sewage treatment facility for treatment prior to being discharged to the environment.

18.3 Tailing and Water Management

18.3.1 Design Basis and Operating Criteria

The principal objectives of the design and operation of the TSF are to provide secure and permanent containment for all tailings solids and potentially acid-generating (PAG)/metal leaching (ML) waste rock, and to provide temporary containment for impounded process water prior to it being transferred to the WMP. The tailings and water management systems are further detailed in the PFS Tailings and Water Management Report (KP, 2021). The PFS design and assumptions are based on the geotechnical conditions of the overburden and bedrock preliminarily assessed based on the information collected during the previous Site Investigation (SI) Programs. Additional SI programs will be required to verify assumptions and support future design work.

Various TSF location alternatives were identified in previous studies completed in 2007, 2010 and 2012. The current TSF location in Cedar Creek, selected by SMG in 2012, was used for the 2012, 2017 and 2019 studies. These previous studies are detailed in the PFS Tailings and Water Management Report (KP, 2021).

The design and operation of the TSF is integrated with the overall water management objectives for the entire mine development, in that surface contact runoff from disturbed catchment areas is controlled, collected, and either contained on site for use in the milling process, or treated and discharged to the environment. The location of the TSF and WMP relative to the other mine facilities is shown on Figure 18-1. An additional requirement for the design and operation of the TSF is to allow for effective reclamation of the tailings impoundment and associated disturbed areas so that post closure land use objectives can be met at the end of the mine operations.

The TSF was sized to store approximately 92 Mt of tailings (based on a mill throughput of 20,000 tonnes per day (tpd) over the 14-year mine life) 66 Mt of PAG waste rock, the Inflow Design Flood (IDF) volume of approximately 10 Mm^3, an operational supernatant pond, plus freeboard. The metallurgical process involves a gravity circuit followed by a rougher flotation circuit to produce rougher tailings. The process feed is reground and subjected to carbon-in-leach and cyanide detoxification circuits before being combined with a pre-float concentrate to produce the cleaner tailings and detox tailings stream, which is assumed to be PAG and ML if allowed to oxidize. The tailings streams will be transported from the plant site to the TSF in separate pipelines. The rougher tailings will be discharged from spigots located at the embankments and the west side of the TSF forming extensive drained tailings beaches. The cleaner tailings and detox tailings stream will be pumped and discharged sub-aqueously into a separate location within the facility, referred to as the PAG Cell. The final dry density of the rougher tailings was assumed to be 1.4 t/m^3. The final dry density of the pyritic tailings was assumed to be 1.3 t/m^3.

Water collecting on the TSF rougher tailings surface will be pumped to the WMP, the primary water management structure. The operating criteria include maintaining the TSF supernatant pond as small as possible. The operating criteria for the PAG Cell include an operating pond over the entire cell to maintain submergence of the PAG tailings and waste rock. The WMP which has been sized to store the site runoff from the 95th percentile wet year assuming a water treatment rate corresponding to approximately the 80th percentile of annual site runoff.

18.3.2 Geotechnical Site Investigations

A geotechnical site investigation was undertaken in 2020 to investigate the subsurface conditions at the TSF and the WMP. The 2020 SI included:

Excavation and logging of 85 test pits.

Drilling and geotechnical logging of 29 drillholes, involving:

- Standard Penetration Testing (SPT) in overburden.
- Hydraulic conductivity packer testing in bedrock.
- Collection of soil and rock samples for laboratory testing.
- Installation of 23 vibrating wire piezometers (VWPs) and dataloggers in eight selected drillholes.

Surficial deposits encountered ranged in depth from 0 m to 30.0 m. The typical overburden profile at the Project site consists of an up to 0.3 m thick layer of organic material overlying well graded glacial till deposits of primarily silty sand, or sandy silt, and hosting varying amounts of clay, gravel, cobbles, and boulders. Potential construction material borrow sources were identified in several locations in the TSF area.

Several fine grained, compact/firm silty sand or clayey silt units were identified interbedded with the glacial till units across the site at depths of up to 26 mbgs. A laboratory testing program which will define the index properties and materials strengths for the overburden tested materials was in progress at the time of this report.

Groundwater levels across the TSF and WMP areas range in depth from ground surface to approximately 10.6 m below ground surface. Artesian conditions were encountered in six drillholes in the TSF area and two drillholes in the WMP.

18.3.3 Waste Management Facility Embankments

The TSF will comprise a north embankment and a south embankment. The embankments will be zoned earthfill/rockfill structures, with a low-permeability core for seepage management. The embankments will include filter and transition zones to ensure proper filter relationships between adjacent zones, and to convey drainage within the embankment. A downstream shell zone, which comprises the majority of the embankment material, will be constructed with Non-PAG mine waste. Cross sections of the north and south embankments are presented in Figure 18-3.

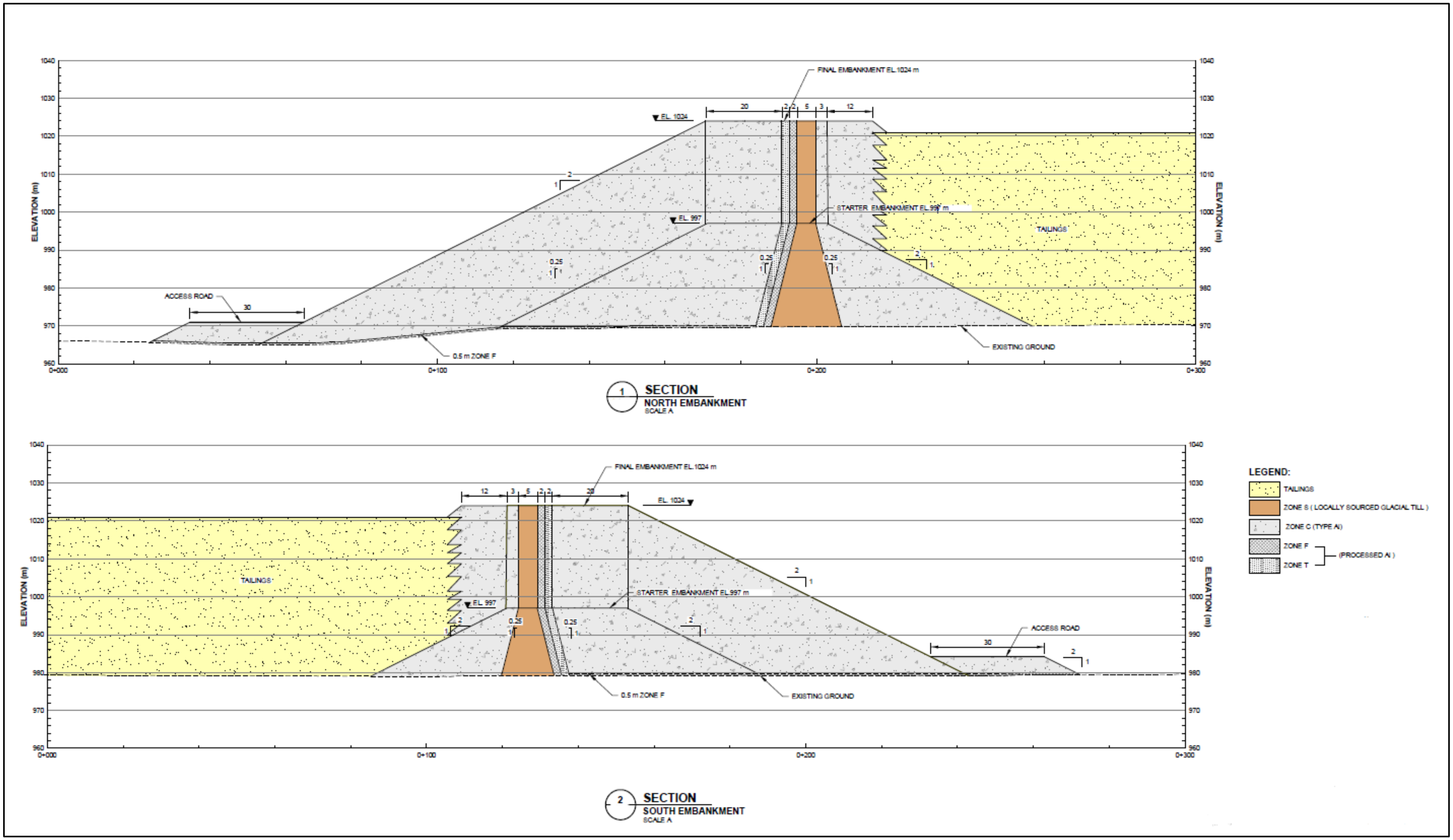


Figure 18-3 Cross sections of the TSF North and South Embankments

The TSF starter embankment, which will be constructed during the pre-production phase, has been sized to store the estimated volume of tailings and PAG/ML waste rock produced during the first two years of operation, plus the supernatant pond volume, the IDF, and associated freeboard allowances. The TSF embankments will be expanded in stages throughout the mine life using the centerline construction method, with each stage providing the required capacity for the period until the next stage of construction is completed. The TSF north embankment will be approximately 31 m high at the starter configuration and 58 m high at its ultimate height. The TSF south embankment will be approximately 17 m at the starter configuration and 44 m high at its ultimate height.

18.3.4 Construction Materials

The TSF embankments will be constructed using suitable waste rock and overburden (low permeability glacial till) from the open pit stripping activities or from local borrow. The embankment zones will be constructed in horizontal lifts, with material for each lift dumped, spread, and compacted in specific lift thicknesses, depending on the material type, to meet the required compacted density requirements. The TSF embankment includes the following zones:

Core Zone – Zone S: The embankment design includes a low permeability core zone (Zone S) for the entire section of the embankment. The core zone will be constructed with well graded low permeability glacial till sourced material from a local borrow or pit stripping. The core zone, in conjunction with the low permeability tailings mass, will function as the primary seepage control barrier.

Filter/Transition Zone – Zone F/T: The transition zone refers to a sequence of processed sand and sandy gravel materials that provide proper filter relationships between adjacent fill zones and control drainage within the embankment. The transition zone materials will comprise at least two zones, including a sand filter (Zone F) immediately downgradient of the core zone, and a gravelly transition zone (Zone T) between the Zone F filter and the downgradient Zone C shell materials.

Shell Zones – Zone C: The shell zones comprise most of the embankment fill material and will be constructed from well-graded Non-PAG waste rock obtained from pit stripping.

Upstream Shell Zone – Zone U: Zone U forms the upstream shell zone and provides the upstream support of the embankment required for centerline construction. The upstream shell zone will be constructed from well-graded Non-PAG waste rock sourced from a pit stripping. The upstream edge of the Zone U material will extend onto the tailings beach consistent with centerline construction dam design.

The internal PAG Cell will be constructed with the PAG/ML waste from the pit stripping activities. The PAG Cell embankment materials will be progressively encapsulated with the bulk tailings as the facility expands. The PAG Cell includes a low permeability natural liner, constructed using glacial till materials, for storage of the PAG tailings and the “Type C” PAG waste rock that requires short term subaqueous disposal.

18.3.5 Tailings Distribution Systems

Tailings from the Process Plant will be delivered to the TSF in two different streams, a rougher tailings stream and a cleaner and detox tailings stream. The rougher tailings distribution system conveys tailings from the Process Plant for discharge into the TSF from the North and South Embankments, and the west side of the TSF, throughout operations. The rougher tailings will be discharged into the TSF from a series of large diameter valved off-takes located along the embankments. The cleaner tailings (together with the cyanide destruction residue from concentrate leaching) will be discharged separately to allow subaqueous deposition within the internal PAG Cell.

18.3.6 Reclaim System

The TSF has two reclaim water systems; one for the rougher tailings area and one for the PAG Cell. The reclaim water for use in the mill processes will be pumped via an HDPE pipe from floating barges on the TSF and the PAG Cell to the WMP. The reclaim water will subsequently be pumped via an HDPE pipe from a secondary floating barge on the WMP to the mill building.

18.3.7 Water Management

18.3.7.1 Water Management Overview

The WMP will serve as the primary site water management component during operations, providing storage for process water, direct precipitation, and runoff. Site runoff, including the water in the separate TSF supernatant ponds, is pumped to the WMP. This significantly reduces the volume of free water stored within the TSF. The WMP provides a buffer for the water treatment plant to reduce the peak flows requiring treatment. The WMP has the capacity to store:

- Mm^3 from the 95th percentile water balance volume,
- Runoff from the 1 in 1,000-year return period storm event from the upstream catchment,
- Approximately 800,000 m^3 assuming the water treatment system is not operating for one month.

An emergency spillway will be constructed at the right abutment to safely pass the peak flows from extreme storm events above the 1,000 year 24-hour storm event.

The WMP will be a fully lined pond with an HDPE geomembrane liner. The embankment will be constructed primarily from Type Ai waste from the pit stripping activities. The upstream embankment face and the basin include a Zone S bedding layer to facilitate liner installation.

Temporary and permanent site water diversion and collection ditches will be used during construction, operations, and closure to minimize sediment mobilization and erosion, collect, and convey mine contact water, and protect natural drainages and watercourses. Runoff from catchments directly upstream of the TSF will be diverted to Cedar Creek, while runoff from catchments upstream of the south embankment will be diverted to Boswell Lake, where it will be pumped around the west side of the TSF to be discharged into Cedar Creek during initial operations. In Year 10, the Boswell Lake South Diversion Channel will be constructed to redirect the Boswell Lake catchment towards the south.

Sediment and erosion control measures will be necessary to limit effects on the surrounding environment and water sources due to earth-moving activities related to the construction and operations of the Project. Common construction activities that have the potential to expose soils to erosive forces include, but are not limited to:

- Clearing vegetation and soil,

- Excavations,
- Blasting,
- Road and trail construction,
- Watercourse diversions or crossings,
- Stockpiling material, and
- Runoff from active work areas.

The Project will take a proactive approach to sediment and erosion control by:

- Limiting the area of exposed soils by minimizing vegetation clearing,
- Installing erosion and sediment control mitigations before construction begins,
- Directing runoff and surface water around active construction and operation areas, and
- Progressively revegetate disturbed areas to improve soil stability and reduce erodible surfaces.

Seepage collection ponds (SCPs) and pumping systems are included downstream of each of the embankments to collect runoff and seepage from the embankments. Water from the seepage collection ponds will be pumped back to the TSF. SCPs have been designed to store a 1 in 200-year return period 24-hr rainfall event, plus an operational pond volume of 2,000 m^3 (an estimate of a minimum-allowed volume, assuming the ponds are kept operationally pumped down), plus a one-meter freeboard allowance. The ponds will include spillways to pass flows exceeding the design storm event.

18.3.7.2 Water Balance Model

The site water balance was developed using a probabilistic model using GoldSimTM software and simulates the supply and demand of water monthly to inform design of the various mine waste and water management components of the mine site. A 52-year long-term synthetic temperature and precipitation record was developed for the model using historic climate conditions calibrated to the Project location. The time-series data were incrementally stepped by year within the model for the life of the Project, preserving the inherently cyclical nature of the temperature record to introduce climate variability into the model. This allowed the water management strategy and the water balance model to be evaluated under a variety of climatic conditions.

The Water Balance Model was used to assess process water requirements throughout the mine life, as well as to simulate the major mine facilities water supply and demand. The results indicate that the site will operate in a surplus water condition during all phases of the mine life and under the full range of variable climatic conditions, including prolonged wet and dry cycles. A schematic of the Water Balance Model for Years 1 through 14 is included as Figure 18-4. The Water Balance Model, including additional period schematics, is included summarized in the Prefeasibility Study Water Balance Model Report (KP, 2021a).

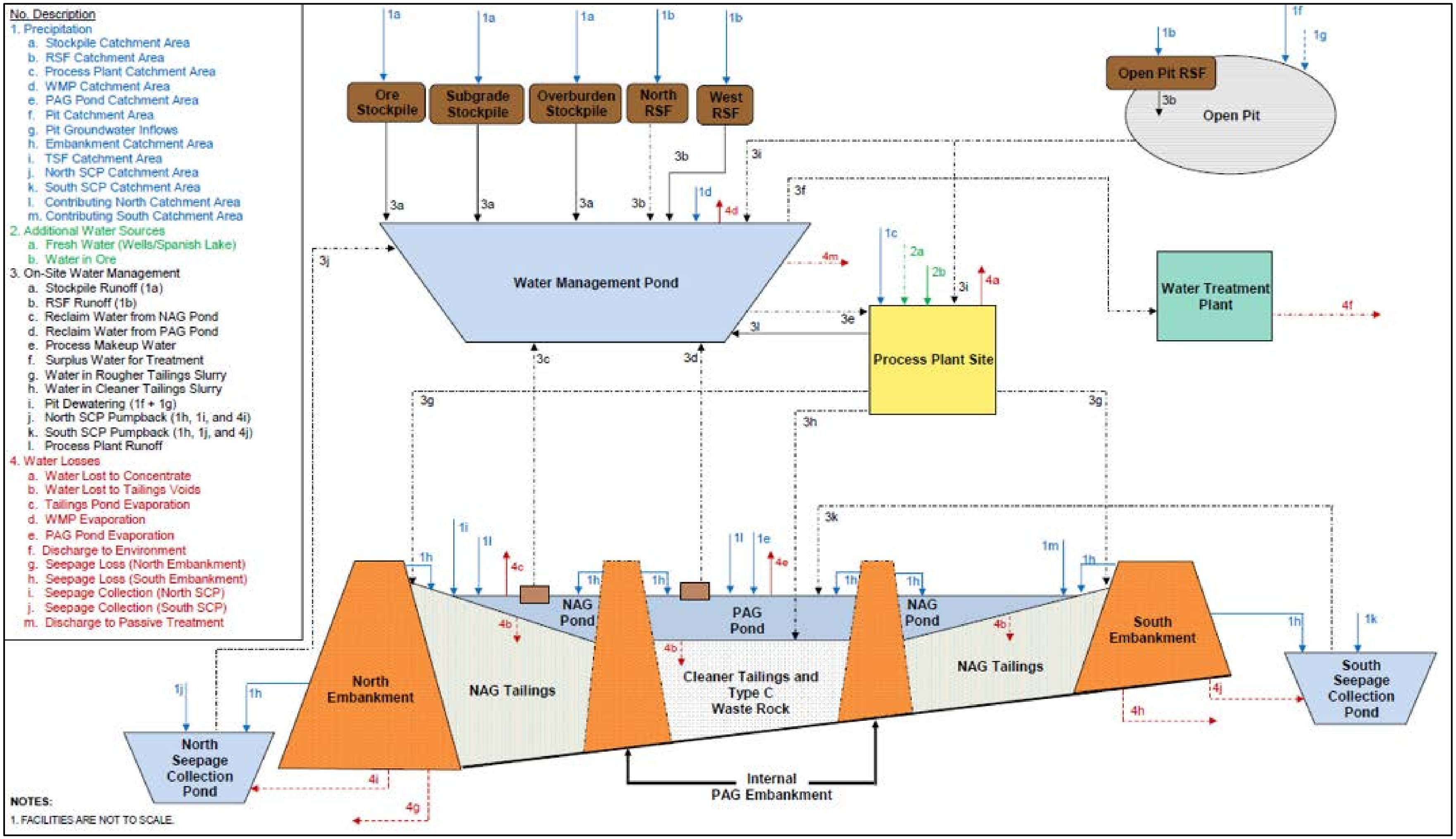


Figure 18-4 Year 1 through 14 Water Balance Schematic

18.3.8 Closure

The mine plan includes four years of active closure, followed by three years of passive closure. The active closure period includes:

- Constructing a closure spillway at the TSF north embankment.
- Re-grading the tailings surface to encourage natural drainage towards the closure spillway at the north embankment. Selective discharge of tailings will occur in the latter years of operations to grade the TSF surface to the maximum practicable extent.
- Construction a closure cover and revegetating the regraded tailings surface.
- Construction of a wetland around the PAG cell.
- Reclamation and revegetation of the TSF embankments.
- Revegetation of the WRSFs which have not been progressively reclaimed during operations.
- Decommissioning and reclamation of the Water Management Pond
- Decommissioning and reclamation of the process plant.
- Progressive reclamation and decommissioning of roads, ditching, and other structures when no longer needed.
- Operation of the passive water treatment systems (by others)

The passive closure phase includes:

- Directing water from the TSF North Embankment SCP to a passive treatment system (by others).
- Directing water from the TSF South Embankment SCP to the Boswell Lake south diversion channel where it will discharge to the south via Winkley Creek.
- Passively treating water in the Open Pit lake within the pit prior to discharge to the environment via gravity drainage.
- Discharge of runoff from the TSF catchment through the TSF closure spillway towards Cedar Creek.

18.3.9 Water Treatment

A WTP will provide a treatment system as part of the ongoing operations, to handle the flows and loads from the process operations, mining operations, and adjacent catchment areas. Water will be received from the Water Management Pond, treated, then once it has met provincial and federal standards, it will be pumped in a pipeline to Cedar Creek and discharged into the environment.

The treatment of MIW will be required during all stages of the Project. During the initial Pre-Production year (Y-2), SMG will manage the suspended solids generated during earthwork and construction through best practices and water management; active treatment is not required in the first year. In the second Pre-Production Year, Y-1, active treatment will be required to treat nitrogen compounds derived from blasting used to generate clean rock for construction; SMG will continue to manage suspended solids without active treatment. Beginning Production Year 1 (Y1) and continuing through active closure (Y15 through Y18), active treatment of MIW will be required. In the active closure period, active treatment will transition to the passive treatment of MIW.

SMG will construct a WTP to actively treat influent from the WMP, which will receive and store MIW generated from the active portions of the Site. The WTP will incorporate a variety of process technologies including oxidation, settling and clarification, microfiltration (MF), denitrification, and RO. The WTP effluent will discharge to Cedar Creek. Design and construction of a 15,000 m^3/day WTP will occur in Y-2, with operations beginning in Y-1. The WTP will initially be equipped to treat nitrogen compounds (blasting residuals) in Y-1. In Y1, the WTP will be fitted with additional treatment equipment to address MIW generated during mining including removal of parameters such as sulphate and metals from the WMP influent. The WTP capacity will be expanded to 18,000 m^3/day in Y6. The WTP will continue to operate through the active closure period (Y15 through Y18). Active treatment will cease in Y18 and the WTP will be dismantled and transported offsite in late Y18.

Separate PTSs will be constructed to address seepage from the following specific sources: The North and West Rock Storage Facilities (RSFs), the North Seepage Collection Pond (SCP), and the Pit Lake. The PTSs will include combinations of iron terraces, BCRs, and aerobic polishing wetlands. Effluent from the West RSF and North SCP PTSs will be directed to Cedar Creek, while that from the North RSF will be directed to Spanish Creek. Primary treatment of the Pit Lake will occur in-situ, with a downstream iron terrace treatment unit for polishing before discharging treated Pit Lake water to Spanish Creek. Design of the North RSF, West RSF, and North SCP PTSs will occur in Y4 and Y5, with construction occurring in Y6 and commissioning beginning in Y7. Design of the Pit Lake PTS will occur in Y11 and Y12, with construction occurring in Y13. The timing of the Pit Lake PTS construction is considered conservative because the Pit Lake will not fill its spill point into Spanish Creek until Y25.

18.3.9.1 Active Treatment

MIW from the Ore Stockpile, Mineralized Stockpile, Overburden Stockpile, North, South, and Open Pit RSFs, Open Pit, NPAG and SAG Ponds, and the North SCP will be collected and routed to the WMP during the life of the mine. Water stored in the WMP will be used in processing. SMG will actively treat excess water in the WMP at the WTP and discharge it to Cedar Creek per a future Effluent Permit under the *Environmental Management Act*.

MIW quality was predicted based on previous site work, water quality from surrogate sites, geology, and modeling. The following parameters are predicted to be Constituents of Concern (COCs) in the MIW: sulphate, arsenic, cadmium, cobalt, molybdenum, nickel, and selenium. In addition, zinc, uranium, and antimony are possible COCs. The SMG Project COC concentration levels were predicted based on data obtained from other existing mines with similar ore and waste rock geochemistry in which ore was processed with carbon in leach (CIL) cyanidation processes. The estimated SMG Project concentrations were also informed by historical humidity cell and barrel test data. Assumed ammonia, nitrate, and nitrite concentrations derived from explosive residues and cyanide destruction were based on confidential and public domain data and modeling.

Likely discharge requirements for the WTP are defined in the British Columbia's Ministry of Environment & Climate Change (2019) *British Columbia Approved Water Quality Guidelines: Aquatic Life, Wildlife & Agriculture*. Where province-specific potential limits were not available, other sources such as the Canadian Council of Ministers of the Environment (CCME) *Water Quality Guidelines for the Protection of*

Aquatic Life were considered. SMG conservatively assumed that no assimilative capacity is available in Cedar Creek; consequently, it is assumed that discharge requirements must be met at the end of the pipe. Average hardness values from surface water samples collected from Cedar Creek in 2020 were used to estimate potential limits for those constituents that are hardness dependent. Table 18-1 summarizes assumed COC levels for the WTP feed water (influent) and target discharge requirements in the effluent.

Table 18-1 Estimated WTP Influent Concentrations and Potential Discharge Requirements

Constituent	Estimated WTP Feed Water Concentration	Potential Discharge Requirement[1]
Sulphate[2] (mg/L)	670	309
Arsenic (µg/L)	54	5
Cadmium[2] (µg/L)	0.6	0.216
Cobalt (µg/L)	3.3	4
Molybdenum (mg/L)	0.2	1
Nickel[2,3] (mg/L)	1.0	0.098
Selenium (µg/L)	24	2
Zinc[2] (µg/L)	420	17.3
Uranium[3] (µg/L)	13	15
Antimony (µg/L)	0.5	-
WAD Cyanide (mg/L)	3.7	5
Ammonia[4] (mg/L)	5.4	1.85
Nitrate[5] (mg/L)	29.5	3
Nitrite[5] (mg/L)	0.6	0.02

Source: Linkan Engineering (2021)

Notes:

1. *2020 average Cedar Creek hardness of 103.1 mg/L as $CaCO_3$ (n =3)*
2. *Hardness dependent variable*
3. *CCME value*
4. *Assumed pH of 7.3 and temperature of 10ºC from 2020 data*
5. *2020 chloride concentrations < 2 mg/L*

SMG will manage the suspended solids in site waters generated during construction in the first Pre-Production Year (Y-2) through best practices and water management; active treatment is not required. In the second Pre-Production Year, Y-1, the WTP will only be required to treat nitrogen compounds derived from blasting residuals used to generate clean rock for construction; SMG will continue to proactively manage suspended solids. Beginning Production Year 1, Y1, and continuing through active closure (Y18), the WTP will treat all COCs shown in Table 18-2.

The WTP is sized to initially treat an average of 15,000 m3/day from the WMP. The plant capacity will later be increased to treat 18,000 m3/day beginning Production Year 6, Y6, to handle larger flows as the mine's footprint expands. During active closure (Y15 through Y18), the amount of MIW treated at the WTP will decrease as a larger percentage of the site water treatment needs is addressed through passive treatment. Active treatment will not be required beyond year Y18. Table 18-2 summarizes the daily average amount of water treated at the WTP and the COCs targeted for treatment.

Table 18-2 Yearly Treatment Objectives

Mine Year	WTP Treatment Rate (m^3/day)	COCs to Treat
Y-2	0	None[1]
Y-1	15,000	Nitrates[1]
Y1 through Y5	15,000	All
Y6 through Y14	18,000	All
Y15[2]	17,050	All
Y16[2]	15,150	All
Y17[2]	14,200	All
Y18[2]	14,200	All

Source: Linkan Engineering (2021)

Notes:

1. *Suspended solids managed through best practices and water management (active treatment not required).*
2. *Active treatment will transition to passive treatment through the active closure phases.*

18.3.9.2 Passive Treatment

MIW from the following three sources will require treatment in closure:

1. Seepage from the north and west rock storage facilities (RSFs).
2. Flow from the north seepage collection pond (SCP).
3. MIW in the main Pit Lake which will receive direct precipitation, groundwater inflows (assumed to be clean), and MIW runoff from the pit walls and the pit RSF.

The PTSs associated with the two RSFs and the north SCP will be constructed during production to assist with water treatment requirements. During closure, seepage from the south SCP will be diverted to Boswell Lake where it will be diluted and polished with natural attenuation mechanisms and is not projected to require treatment.

The flows to be treated from the RSFs, SCPs, and the pit lake are summarized in Table 18-3.

Table 18-3 Assumed Flow Rates at Closure

MIW Source		Assumed Flow to be Passively Treated	
		(L/min)	(m^3/day)
	North RSF	487	701
	West RSF	902	1,299
	North SCP	1,250	1,800
	South SCP	0	0
Pit Lake Outfall		3,125	4,500

The Passive Treatment layout is indicated in Figure 18-5.

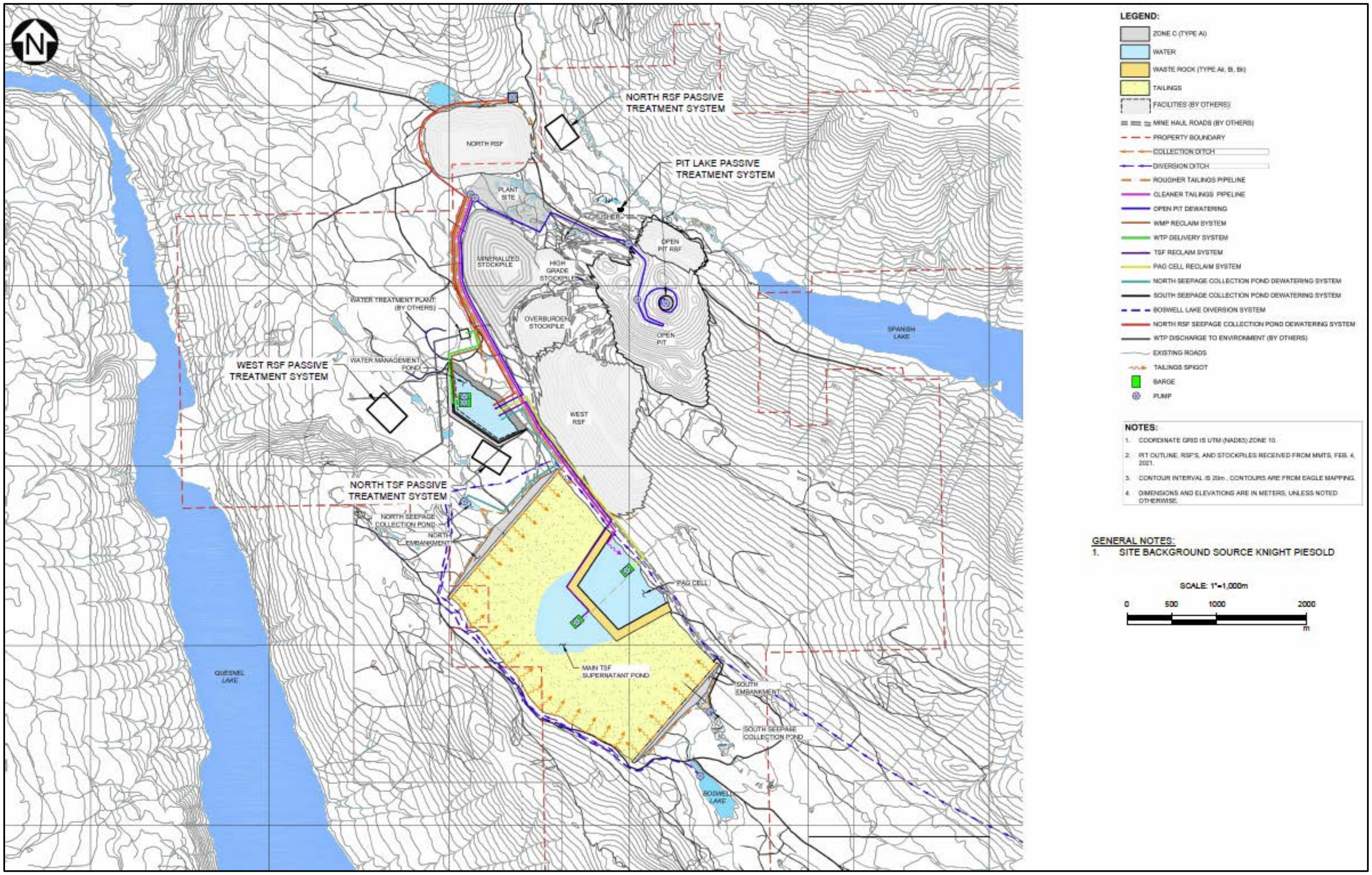


Figure 18-5 Passive Treatment Layout

Based on the influent flow rates and chemistry, and the potentially applicable discharge requirements, the following potential passive treatment processes were identified for use in the closure phase at the Spanish Mountain Gold Project:

- Iron terrace for arsenic removal and ammonia oxidation to nitrate.
- Biochemical reactor (BCR) for nitrate, sulphate, and most heavy metal removal (as sulphides).
- Aerobic polishing wetland (for minor removal of biochemical oxygen demand).
- In situ pit lake treatment.

18.4 Off-Site Infrastructure

The Spanish Mountain site will be served at 138 kV from BC Hydro.

18.4.1 138 kV Source Substation

BC Hydro will establish a new 138 kV Substation near Highway 97, to be known as SMMX Substation. The new substation will be fed at 230 kV from existing BC Hydro line 2L95. The new substation will contain a 30 MVA step-down transformer to 138 kV, and metering at 138 kV. A set of outgoing suspension insulators will form the Point of Interconnection (POI). The new substation will be owned and operated by BC Hydro. Everything downstream of the POI will be owned and operated by Spanish Mountain.

18.4.2 138 kV Transmission Line

A new 138 kV transmission line will be installed between the POI and the receiving substation at the Spanish Mountain mine site. The transmission line will be single pole type. The transmission line will generally follow the road, to a point just west of the town of Likely, then it will be routed north, around Likely, to the mine site. The transmission line will be owned and operated by Spanish Mountain.

18.5 On-Site Infrastructure

The on-site infrastructure consists of an incoming 138 kV substation, and electrical distribution at 13.8 kV on-site.

18.5.1 138 kV Receiving Substation

The 138 kV substation will consist of:

- An incoming structure complete with 138 kV disconnect switch
- An incoming 138 kV SF6 circuit breaker
- A 30/40/50 MVA transformer, 138 kV/13.8 kV
- A building containing 13.8 circuit breakers
- A lineup of 13.8 distribution circuit breakers
- A 13.8 kV capacitor bank

18.5.2 13.8 kV Distribution

The 13.8 kV distribution will be housed inside a building within the substation. The 13.8 kV circuit breaker lineup will consist of 2 high units. The distribution to the Process Area will consist of:

- A 13.8 kV feeder (3C#250MCM) to a 7.5 MVA, 13.8 kV/4.16 kV transformer serving the Mills Area
- A 13.8 kV feeder (3 runs of 3C#250MCM) to a 13.8 kV bus serving the SAG and Ball Mill
- A 13.8 kV feeder (3C#250MCM) to a 1.5 MVA, 13.8 kV/4.16 kV transformer serving the Crushing Area
- A 13.8 kV feeder (3C#250MCM) to a 2 MVA, 13.8 kV/4.16 kV transformer serving the Pebble Crusher Area
- A 13.8 kV feeder (3C#250MCM) to a 10 MVA, 13.8 kV/4.16 kV transformer serving the Reagents/CIL Area
- A 13.8 kV feeder (3C#250MCM) to an overhead distribution line feeding all loads outside of the Process Building (see below)
- A 13.8 kV feeder (3C#250MCM) feeding a capacitor bank located within the substation
- A spare 13.8 kV circuit breaker

Single-line diagrams and Transmission Line layout route is presented in Appendix B.

18.5.3 13.8 kV Overhead Distribution

The overhead 13.8 kV line will service the following area:

- Maintenance and Truck Shop
- Waste and Sewage Systems
- Administration/Dry Building
- Assay Lab
- Cold Storage Warehouse
- Explosives Manufacturing and Storage
- Water Treatment Plant
- Tailings/Reclaim Systems

19 MARKET STUDIES AND CONTRACTS

The Project will yield gold doré as its final product, which is expected to be sold on the spot market through marketing experts retained by SMG. Spanish Mountain Gold has not completed any formal marketing studies regarding gold and silver production that will result from the mining and processing of ore from the Spanish Mountain Gold Mine into doré bars.

Gold can be readily sold on numerous markets throughout the world; its market price at any time is easily and reliably ascertained. The large number of available gold purchasers, both domestically and internationally, allow for gold production to be sold on a regular and predictable basis, and on a competitive basis with respect to the spot price. As of late March 2021, the average consensus price forecast from 30 investment dealers estimated a gold price of US$1,765/oz in 2023, US$1,712 in 2024 and US$1,599/oz over the long term.

A gold price of US$1,600 per ounce, and a silver price of US$24 per ounce, have been used for the 2021 Spanish Mountain Gold Prefeasibility Study. Spanish Mountain Gold plans to contract out the transportation, security, insurance, and refining of doré. Smelter terms / offsite costs related to the sale of doré and used for the study are shown in Table 22-1. An exchange rate of C$1.00:US$0.76 was used, based on the trailing three years C$:US$ foreign exchange rate.

The QP, Marc Schulte, P.Eng., expects that terms contained within any potential sales contract would be typical of, and consistent with, standard industry practices. The assumed market prices, exchange rate, and smelter terms/offsite costs are considered reasonable with respect to the current market and can be used to support the project economic analysis.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

Throughout its history, Spanish Mountain Gold Ltd (SMG) has made considerable efforts to undertake environmental studies and community engagement to facilitate the advancement of the project. The following presents a summary of the environmental aspects, permitting and social or community impacts of the work program to date. Much of the environmental and social engagement activity was completed between 2007 and 2012, to support an earlier mine plan. SMG will capitalize on the successes of that program as it is re-invigorated in 2021. All environmental programs have been re-initiated in 2020 and are continuing in 2021.

20.1 Environmental Studies

20.1.1 Meteorology

An industry standard climate station was installed at the Spanish Mountain exploration camp in August 2010. The estimated mean monthly precipitation, temperature, and evaporation distribution for the Project are presented in Table 20-1. Precipitation and temperature values are based on measurements from the site climate station, as well as from sites operated near to the Project. Average monthly conditions were used to estimate typical evaporation rates using the Thornthwaite Equation. The values provided are for an elevation of 1,080 masl; the elevation at the Project site ranges between 900 masl and 1,300 masl.

Table 20-1 Mean Monthly Climatic Parameters

Parameter	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
Precipitation (mm)	81	46	38	45	73	120	90	89	67	69	73	71	860
Temperature (°C)	-7.9	-4.9	-0.1	4.8	9.6	13.1	15.9	15.0	10.4	4.2	-2.1	-7.0	4.2
Evaporation (mm)	0	0	2	34	71	94	113	99	62	25	0	0	500

20.1.2 Air Quality

KP installed passive dustfall stations at five locations in 2010. Each station was comprised of two pre-washed HDPE containers set in separate windscreens, with the top of the containers and windscreen set 2 meters above ground surface, in accordance with Standard Test Method for Collection and Measurement of Dustfall (Settleable Particulate Matter) ASTM D1739 - 98. One container was analyzed for total dustfall (soluble and insoluble) and the other was analyzed for total metals, both measured in units of mg/dm^2-day. Fugitive dustfall samples were collected approximately monthly from October 2010 to November 2012.

Results were typical for a largely forested area in the baseline condition, with generally low fugitive particulates and concomitantly low metals fraction.

20.1.3 Surface Hydrology

The Spanish Mountain Project lies within the Spanish Creek watershed, which is a tributary to the Cariboo River. Small ephemeral and unnamed creeks flow through the Spanish Mountain Project site, and into Spanish Creek, below Spanish Lake. Project facilities will also be located in the Cedar Creek watershed, which drains directly to Quesnel Lake.

The creeks in the Spanish Mountain Project area are generally characterised by high flows in the spring due to snowmelt, and rainfall combined with snowmelt, medium flows in the late summer/ fall, and very low flows in the winter. Generally, Spanish Creek and Cedar Creek flow continually throughout the year, but flows can be affected by intermittent ice formation during winter.

Five automated hydrology measurement stations were active historically in the Project area; all of which were re-installed in 2020. Locations of each station are illustrated on Figure 20-1. Ongoing database development continues to inform the surface hydrology program.

Historically, freshet flows in Spanish Creek range between 4 and 8 m^3/s, and low flow periods occur in the late summer/early fall or during mid winter, with values between 0.15 and 0.40 m^3/s.

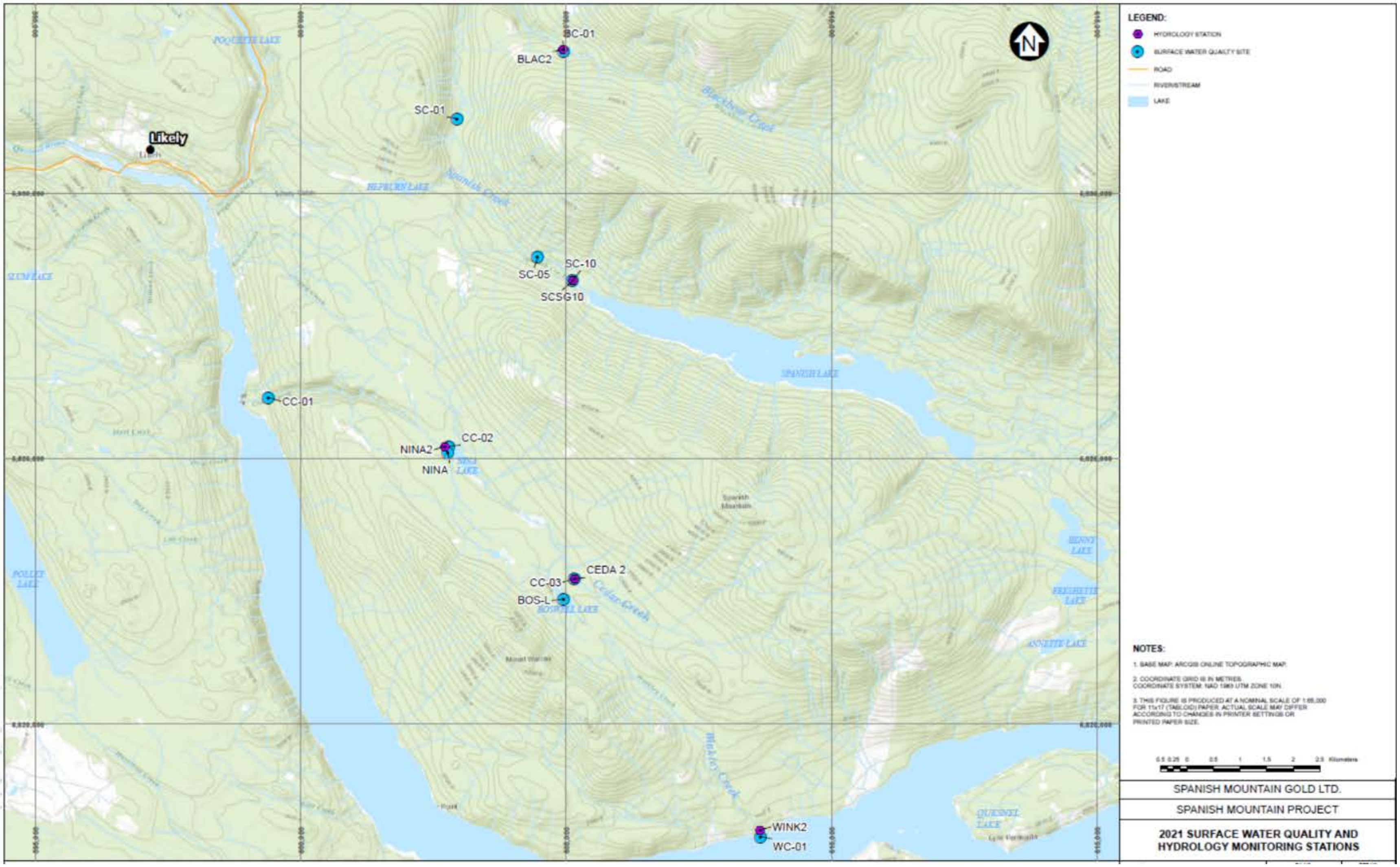


Figure 20-1 2021 Surface Hydrology and Water Quality Monitoring Locations

20.1.4 Surface Water Quality

A total of 28 stream and lake sites were established around the Spanish Mountain Project area, starting in 2007, and monitored until the fall of 2012; surface seeps were also monitored intermittently as part of the surface water program during this period. Water quality samples were collected from 10 of the established monitoring sites in September 2020 as illustrated on Figure 20-1. Water quality samples from within the claim boundary have consistently shown concentrations of total and dissolved metals that exceed limits set by the CCME and the BC Water Quality Guidelines (BCWQG) for the protection of aquatic life. The level of these concentrations is likely caused by the natural mineralogy of the claim area and historic placer mining activities.

20.1.5 Groundwater Quality and Hydraulic Conductivity

Two groundwater monitoring wells were installed in the Project area in 2009 (one near Nina Lake and the other downstream from Spanish Lake), and one water sample was collected from each in October 2009. SMG retained Knight Piésold Ltd (KP) to carry out a hydrogeological site investigation in 2011 to develop baseline groundwater conditions and support ongoing design studies. Sixteen wells were installed at nine locations as part of the site investigation, with the wells located primarily around the location of the then-proposed tailings storage facility to collect geological information, on-going water levels and groundwater quality information to aid with characterizing baseline conditions to support monitoring and contingency planning as mine planning proceeded. Response tests with reliable data output were completed on 13 of the 16 wells. The estimated hydraulic conductivity was between $2.4\text{x}10^{-8}$ m/s and $2.0\text{x}10^{-5}$ m/s. The hydraulic conductivity in the wells installed in the overburden was between $8.8\text{x}10^{-6}$ m/s and $9.0\text{x}10^{-5}$ m/s. KP was retained by SMG in 2012 to conduct an additional hydrogeological site investigation program downgradient of the waste management area proposed at the time. Six groundwater wells were installed as part of this program. All 2012 monitoring wells were installed in bedrock and had estimated hydraulic conductivities ranging between 2.0×10^{-8} m/s and 7.0×10^{-6} m/s.

Water quality samples were collected from these wells between October 2011 and September 2012. Several of the samples had metal concentrations that exceeded provincial freshwater aquatic life guidelines, including arsenic, copper, iron, manganese, selenium, and zinc.

20.1.6 Fish and Fish Habitat

KP conducted fish and fish habitat studies from between 2007 and 2012 in the Project area. Rainbow trout were the only species captured using minnow trapping and electrofishing in Spanish Creek. Dace, burbot, rainbow trout, and chinook salmon were captured in Cedar Creek near the confluence with Quesnel Lake, while rainbow trout was the only fish species captured at the upper sites in Cedar Creek and all other sites.

More recent fish and fish habitat surveys were conducted in September 2020 to assess presence/not detected of fish species in the waterbodies around the proposed mine area and assess any changes in fish habitat and access to the monitoring sites previously visited between 2007 and 2012. Fish species distribution was consistent with the previous surveys.

20.1.7 Vegetation and Wildlife

The Project is within the area under the Horsefly Sustainable Resource Management Plan (SRMP), which is part of the implementation of the Cariboo Chilcotin Land Use Plan. The Horsefly SRMP provides area-based resource targets and strategies for timber, range, mining, fish, wildlife, biodiversity conservation, water management, tourism, recreation, agriculture, and wildcraft/agro-forestry. Key wildlife and vegetation criteria include Grizzly Habitat; Old Growth Management Areas; Ungulate Winter Range; Wildlife Habitat Areas; and Moose Wetlands.

Field surveys conducted between 2010 and 2012 were used to confirm the presence of focal wildlife and vegetation species and ecosystems, confirm habitat associations of focal species, and confirm habitat characteristics and accuracy of the typed ecosystem polygons within mapped areas. Wildlife and habitat surveys were conducted according to provincial standards documented by the Resources Information Standards Committee.

Detailed Terrestrial Ecosystem Mapping (TEM) was prepared to model wildlife habitat. Mapping was completed at a 1:10,000 scale. Most of the Project is located in the Quesnel variant of the Interior cedar-hemlock wet cool subzone with some Cariboo variant Engelmann spruce-subalpine fir wet cool subzone at higher elevations (Steen and Coupé, 1997). Common tree species are western red cedar, spruce, hemlock, subalpine fir, Douglas-fir, lodgepole pine, birch, and various poplar species. The area has a history of disturbance and there is limited old forest remaining.

Amphibian surveys identified Columbia spotted frog, pacific chorus frog, western toad, wood frog and long-toed salamander. Western toad is a Species at Risk Act (SARA) Schedule 1 species. Numerous bird species were detected, including Olive-sided flycatcher (SARA Schedule 1, and Blue listed provincially). Acoustic surveys identified 11 species of bats.

Winter snow tracking transects for ungulates and furbearers were conducted, with transects selected to sample various habitats in different Project areas, with a focus on older forest. Results from the transects indicated little use of the area except by moose; other species detected included deer, wolf, coyote, lynx, marten, snowshoe hare, and squirrel.

Previous discussions with government regulatory agencies were undertaken to develop methods to avoid or mitigate negative environmental effects. None of the environmental parameters identified to-date are expected to have a material impact on the ability to extract the Mineral Resources or reserves. All baseline environmental programs have been re-initiated in 2020 and are continuing in 2021.

20.1.8 Heritage

An updated Archaeological Impact Assessment was conducted throughout the entire mineral concession in 2019 (Terra Archaeology, 2019). It confirmed that the Project is unlikely to impact heritage resources.

20.2 Permitting

The Environmental Assessment (EA) process began on July 8, 2011, with the submission of a project description to the BC Environmental Assessment Office (EAO) and the federal Canadian Environmental

Assessment Agency (CEA Agency). Detailed environmental and socio-economic baseline studies were then initiated and conducted over a two-year period. Advancement on the EA was halted by SMG in 2012 while project design updates were completed. Between 2012 and 2019, both provincial and federal processes were kept open, with SMG providing annual updates to the EAO regarding its intention to continue the EA process. The Project was formally terminated from the federal process on August 28, 2019, as new legislation was brought into force. SMG withdrew from the provincial process on December 20, 2019.

Both levels of government enacted new environmental assessment legislation, which came into force in 2019. The Project will be required to comply with the new legislation as it re-enters the EA process. SMG will have the opportunity to work with the EAO and CEA Agency to utilize the work completed in previous years as much as possible to move the process forward.

Although the nomenclature of each step in the provincial and federal EA processes differ, they largely follow similar procedures, which allows for submission to be catered to both levels of government simultaneously. In the provincial process, the substantive document that summarizes all baseline information, assesses impacts, and provides management and monitoring plans is referred to as the Application. This document in the federal process is called the Impact Statement. Upon successful completion of the provincial process, an EA Certificate is issued that specifies conditions within which the project can proceed. The federal process culminates with a Decision Statement, which also specifies conditions. In both processes, the EA must consider biophysical, social, cultural, and health factors, including climate change and sustainability. Both processes rely heavily on review and comment from First Nations and the public.

Upon completion of the provincial and federal EAs, SMG and consultants will then work with provincial and federal regulators to advance the required permits and authorizations. Major mines require several authorizations from many different provincial and federal agencies. The key permits are expected to be:

- Provincial Mines Act permit
- Provincial waste discharge authorizations under the Environmental Management Act
- Federal Fisheries Act Authorizations
- Amendment to Schedule 2 of the Federal Metal and Diamond Mining Effluent Regulations (MDMER).

Several other permits will also be required, including licences or approvals for water use or water storage facilities under the Water Sustainability Act and the B.C. Dam Safety Regulation, authorizations for any cutting or spoiling of crown trees under the Forest Act, and land tenure under the Land Act.

20.3 Waste and Tailings Disposal, Site Monitoring, and Water Management

Waste and tailings disposal, and their attendant water management strategies are discussed in Section 18.0.

The current Prefeasibility Study is based on the results of a geochemical characterization program undertaken by SRK (SRK Consulting 2012) to assess the acid rock drainage and metal leaching (ARD/ML) potential of waste rock and tailings expected from the proposed open pit development of the Spanish Mountain deposit. The study consisted of composite samples of drill core representing potential waste rock and low-grade ore from four main rock types, as well as available samples of flotation tailings and cyanide leach tailings generated from bench scale metallurgical testing. Static tests included acid-base accounting (ABA), trace element analyses and mineralogical evaluation. Kinetic tests included laboratory humidity cells and field barrels to assess sulphate oxidation rates and metal leaching potential.

Results of the 2012 study stated that the proportion of rock with ARD potential at Spanish Mountain is relatively low. Potential leaching of a number of parameters were identified regardless of ARD potential though arsenic appears to be the most significant. Surrogates for sulphur and carbonate content were identified to support block modelling utilizing the ICP database. The 2012 results have been used for waste management planning. Rougher tailings tested in the study had low potential for ARD and low metal leaching potential. The cyanide leach tailings had a high potential to produce ARD and leach molybdenum, lead, silver, arsenic, cadmium, and selenium.

An expanded geochemical characterization program is currently in progress to support feasibility evaluations and permitting for the project. This expanded program includes additional sampling and testing of drill core representing potential waste rock and low-grade ore from the eight geological domains within the proposed open pit. Geochemical testing is being conducted in a staged approach. In late 2020, static testing was undertaken including ABA and trace element chemistry. Analyses planned for 2021 will include mineralogical characterization, leach extraction testing and kinetic testing including humidity cells and re-starting the original field barrels. In addition, samples of flotation tailings and detoxified cyanide tailings solids and supernatant from pilot plant metallurgical testing will also be characterized.

Site-specific water quality modelling will evaluate the effects of any discharge to surface and groundwater. Containment strategies for the waste material will be implemented to minimize air and water exposure of the reactive waste material. Drainage from waste rock storage areas and mine workings will be monitored for the life of the Project.

20.4 Social or Community Requirements

First Nation and public engagement and consultation represents a critical component of the Project development. SMG has invested significant resources in these endeavours and will have an increased effort as the Project re-enters the environmental assessment process, and throughout the construction, operation, and closure phases of the Project.

20.4.1 First Nations

SMG is committed to working closely with First Nations to maximize community participation and involvement in the Spanish Mountain Gold Project throughout all stages of the mineral exploration and mining cycle.

The Project lies within the following traditional territories:

- Secwepemc Nation Tribal Council

- T'exelcemc First Nation (Williams Lake Indian Band).
- Xat'sull First Nation (Soda Creek Indian Band).

Carrier Chilcotin Tribal Council

- Lhtako Dene Nation (Red Bluff Indian Band).

SMG values early engagement and ongoing dialogue during all stages of the mining cycle and has established collaborative working relationships all three First Nations, drawing on a variety of engagement activities during the previous environmental assessment work. Engagement is focussed on communicating the Project plans to ensure that all members of each First Nation have an opportunity to learn about and participate in the various work planning activities undertaken on the Project. Engagement activities have utilized a spectrum of methods, including:

- Meetings with chiefs and council
- Community meetings and workshops open to all members
- Project tours on site with chiefs and council, as well as elders and community members
- Involvement / contracting of members to conduct environmental studies
- Sponsorship / contracting of Traditional Use Studies of the Project area

SMG maintains signed cooperation agreements with each of the three First Nations. These agreements govern the participation of each party during the EA and permitting review of the Project and are expected to set the foundation of future Project agreements.

It is important to SMG that First Nations are appropriately informed, engaged, and consulted about the Project. Increased consultation with each First Nation is anticipated as the Project moves from Prefeasibility to Feasibility and re-enters the environmental assessment process.

20.4.2 Communities

The Project is located just east of the village of Likely (population 350) in the eastern portion of the Cariboo Regional District. The Project is accessed via the Likely Road, which connects to Highway #97.

The Cariboo Regional District is a regional government consisting of 12 electoral areas (A through L). Each electoral area elects a Director, except for the incorporated member municipalities (Williams Lake (population 10,000), Quesnel (population 10,000), 100 Mile House (population 1,900)), which are represented by an elected Mayor and Council. The Project resides in Cariboo Regional District Electoral Area F, which includes the communities of Likely, Big Lake and Horsefly.

The village of Likely is expected to receive the highest concentration of direct activity during mine-life and will be the community primarily affected by Project activities and related infrastructure. The two other communities located inside electoral area F, Big Lake and Horsefly, will also see project related activity. The Big Lake community sits along the Likely Road transportation corridor while the Horsefly community is located south of Quesnel Lake and is accessed via the Horsefly Road, which connects at Highway #97.

The communities of Quesnel, Williams Lake, and 100 Mile House and the unincorporated communities of McLeese Lake, Lac la Hache, and 150 Mile House, are located along the main regional transportation corridor, Highway #97. These communities are expected to experience both direct and indirect effects from Project activities. It is also anticipated that all these communities may be involved in the sourcing of labour, goods and services for the Project.

Land users with interests overlapping the Project footprint, including trappers, commercial recreation operators, guide outfitters, placer miners, a community forest, and water license holders, have also been identified. SMG initiated communications with land users.

SMG has been informally interacting with community members and businesses by maximizing locally sourced labour and supplies and participating in, and supporting, various community events. Additionally, SMG has previously conducted an array of community consultation activities, which have included:

- Community open houses in Likely, Big Lake, Horsefly, Williams Lake, and Quesnel
- Presentations to Williams Lake mayor and council, CRD, chamber of commerce
- Publication of a Community Newsletter (paper flyers and online)
- Maintenance of a community email to respond to inquiries.

In addition to focussed consultation and engagement activities, SMG has been a participant and sponsor at community events, including the Likely Victoria Day, Likely School field trips, golf tournaments, and fund raising for the local fire hall and community hall.

SMG prioritizes local hiring, and sources supplies and materials from local communities to the extent possible.

20.5 Mine Closure

A mine closure and reclamation plan is required to ensure that developed areas are restored to viable and self-sustaining ecosystems, and that safety and end-use land objectives are met. A detailed closure plan will require more thorough studies that include an environmental evaluation of the mine wastes (dumps and tailings), ultimate pit wall compositions, hydrologic regimes, and end use. These studies are typically completed as part of a feasibility study, and subsequently the Environmental Assessment process and Mines Act permit. SMG will provide financial assurance that reclamation can be completed through posting of a reclamation bond, as required by the Mines Act; SMG will update its closure plan once every five years. A preliminary estimate of the reclamation bond is described in Section 21.0.

To support the Prefeasibility, a high-level closure concept was developed, wherein it was assumed that concurrent reclamation would be conducted as practicable during operations, followed by 4 years of Active Closure activities on cessation of processing. The Active Closure period would include the removal all reagents, explosives, and petrochemicals, deconstruction of the processing infrastructure and all buildings. Useable parts would be sold, and metal or other recyclable materials would be salvaged.

The pit will be stabilized, and the phreatic surface will be allowed to rise to baseline.

The WRSF will be graded, contoured, and revegetated.

During Active Closure, the Tailings Storage Facility will be modified as described below:

- A closure spillway will be constructed at the north embankment.
- The tailings surface will be re-graded to encourage natural drainage towards the closure spillway at the north embankment. Selective discharge of tailings will occur in the latter years of operations to grade the TSF surface to the maximum practicable extent.
- A closure cover will be constructed, and the tailings surface will be revegetated.
- A wetland will be constructed on the surface of the covered TSF in the area of the PAG cell.
- The TSF embankments will be graded and revegetated.

All site water, including drainage from the pit, processing area, and TSF, will be collected in diversion ditches that will report to passive water treatment systems. Construction and operation of the PTSs will be online during operations, except for the open pit which will be developed during the active closure phase. Active pumping from Boswell Lake to Cedar Creek downstream will be curtailed; a South Diversion channel will be constructed, allowing discharge from Boswell Lake to report to Winkely Creek.

Following the Active Closure phase there will be a 3-year Passive Closure phase. During this phase, all remaining drainage will be directed to the PTSs, which will be monitored to ensure proper functioning.

20.6 Comments on Section 20

The environmental and community aspects of the Project have been well studied and are understood. Ongoing and updated studies continue to add clarity to the baseline condition and will be used as a foundation to re-enter the environmental assessment and permitting processes. There are no known environmental or social constraints that could materially impact the issuer’s ability to extract the mineral resources or mineral reserves.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

To complete the cost estimate SMG engaged a team of independent consultants co-ordinated by MMTS. MMTS was responsible for the mining costs; Ausenco, for metallurgy, processing, on-site infrastructure, off-site infrastructure; and KP for the TSF, overall Site Water Management and Environmental Studies, MCA for the main off-site overhead transmission line, receiving substation at site and on-site electrical distribution to non-process related areas.

A consolidated capital cost estimate was prepared for the Project based on open pit mine operations with a process plant with a capacity of 20,000 t/d; supporting infrastructure, 138 kV overhead transmission line, primary 138 kV substation, site power distribution, TSF, and water management; and a WTP. The overall estimate (except for Water Treatment estimates – which is a Class 5 AACE estimate) describes the methodologies and sources of information applied to prepare an estimate that meets the American Association of Cost Engineers (AACE) Class 4 requirement of an accuracy range between -25% to +25%. The purpose of this Section is to summarize the capital costs and describe the methods, assumptions, and exclusions used in developing the estimate.

These cost estimates are expressed in Q1 2021 Canadian dollars ($), with no allowances for price escalation or currency fluctuations. An exchange rate of US$0.763 to C$1.00 has been used for any conversions (except for the Water Treatment estimates which used an exchange rate of US$0.80 to C$1.00). All values are shown in Canadian dollars unless otherwise noted.

The capital cost estimate was divided into initial and sustaining capital. For the estimate, CAPEX was identified as initial capital for all costs before production. All capital expenditure incurred on commencement of production through to the end of the mine was categorized as sustaining capital. Capital Cost Estimate Total Project costs are presented in Table 21-1, for each work breakdown structure (WBS).

SMG benefits from significant existing infrastructure, which helps reduce the initial capital cost. Total initial pre-production capital cost is $607.2M inclusive of construction indirect costs, engineering-procurement-construction-management (EPCM), contingencies and owners' costs. The sustaining capital is $290.5M inclusive of mine development capital, and process plant. The LOM capital expenditure (CAPEX) is $897.7M exclusive of closure costs. Total Closure costs is $159.6 include reclamation of the open pit, process, and non-process facilities, tailing storage facility, surface site water management, environmental items, project indirects Owners costs, and contingencies. Open Pit mining and haulage are anticipated to be completed using an owner-operator development model operating 365 d/a with a mobile equipment fleet which is included in operating costs.

Table 21-1 Total Project Capital Cost Summary by Area

WBS	Description	Total Cost ($ '000s)		
		Initial	Sustaining	LOM Total
1 Direct Costs				
10	Overall Site Development	25,600	1,950	27,550
20	Mining	73,438	163,731	237,169
30	Ore Handling	33,743	-	33,743
40	Process	125,527	8,057	133,584
50	Tailings and Water Management	40,318	35,611	75,929
60	Environmental	1,600	7,200	8,800
70	On-Site Infrastructure	41,617	-	41,617
80	Off-Site Infrastructure	64,170	3,250	67,420
82	Water Treatment Facilities	10,347	35,006	45,353
2 Indirect Costs				
90	Project Indirect Costs	101,964	18,510	120,474
3 Owner's Costs				
98	Owner's Costs	13,654	-	13,654
4 Contingency				
99	Contingency	75,225	17,175	92,400
Total Project Costs		**607,203**	**290,490**	**897,693**

Detailed reports of the capital and sustaining cost estimates has been coded in accordance with the Project Work Breakdown Structure (WBS). Refer to Appendix C for the detailed capital and sustaining capital and the WBS.

Initial capital was designated as all capital expenditures required prior to plant start-up to produce gold and silver doré bars.

The purpose of the capital estimate is to provide substantiated costs which can be used to assess the economics of the project at a prefeasibility level of study. The cost estimate is based on an engineering, procurement, and construction management (EPCM) implementation approach.

21.1.1.1 Work Breakdown Structure

MMTS used the Work Breakdown Structure (WBS) as directed by SMG. This WBS was used by all sub-consultants and SMG to estimate their costs.

The WBS comprises Primary Tags (major, area, and subareas) and Secondary Tags (Bid section and Section)

Each estimate task is coded as follow:

- e.g., 402010-5010-(Y-1)-B07-0312 – Ball mill
- Sub-area-Section-Year of work-worksheet-sequence

- Primary tag is 402010 – this sub-area represents Grinding
- Secondary tag is 5010. This represents new mechanical equipment.
- Construction is Year-1

The worksheet used is B07 with the sequence of 0312 (this number is also the row number of the worksheet) presented in Appendix C-1.

The WBS was used by all contributing parties; it is a hierarchical system based on primary tags (major, area, sub-area) and secondary tags (commodities).

The estimate was prepared using a combination of Excel-based estimate templates and software. All estimate entries were entered in the template, with the data imported. A standard coding system, based on the WBS and commodity codes was used to categorize each entry and organise the estimate.

21.1.2 Mining Capital Costs

Mine capital costs have been derived from vendor quotations and operational data collected by other Canadian open pit mining operations.

Pre-production mine operating costs (i.e., all mine operating costs incurred before mill start-up) are capitalized and included in the capital cost estimate. Pre-production pit operating costs include grade control, drill and blast, load and haul, support, and GME costs. GME covers the salaries and costs for mine operations and engineering staff.

All mine operations site development costs—such as clear and grub, topsoil stripping, wetland removal, haul road construction, and explosive pad preparation—are capitalized. Pit dewatering and depressurization costs are estimated, which includes drilling vertical wells and horizontal holes, pump installations and maintenance.

The mine equipment fleet purchases are planned as financing or lease agreements with the vendors. Down payments and monthly lease payments are capitalized through the initial and sustaining periods of the project.

Equipment pricing is based on new units delivered to the mine, with all transportation, assembly and commissioning costs included. Unit prices for most of the fleet are based on recent vendor budgetary quotations. Others are sourced from the MMTS equipment database. Used equipment, if available, will reduce these equipment capital costs, and have not been considered for this study.

Mine fleet will consist of diesel-powered equipment and no electric power will be required in the pit. Power to operate pumps and depressurization wells will be from diesel generators.

Estimated fleet spare and estimated initial fuel, lube, and tire inventories are capitalized.

The following items are also capitalized through the initial and sustaining periods:

- Explosive’s magazine,
- High precision GPS (global positioning system) and machine guidance systems,

- Fleet management system,
- Communication radios,
- Mine survey gear and supplies,
- Geology, grade control, and mine planning software licenses,
- Maintenance tooling and supplies,
- Mine rescue gear,
- Geotechnical instrumentation,
- Piping for pit dewatering and culverting materials for haul roads

21.1.3 Process and Infrastructure Basis of Capital Cost Estimate

21.1.3.1 Methodology

The estimate is divided into direct and indirect costs before production.

Direct costs are those costs pertaining to the permanent equipment, materials, and labour associated with the physical construction of the facilities.

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, including construction contractor's indirect costs.

21.1.3.2 Source Data

The following information was used in preparing the estimate:

- Mechanical Equipment list;
- Electrical Equipment list;
- 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;
- Budget quotations;
- Benchmarking against similar project.

21.1.3.3 Market Availability

The pricing and delivery information for quoted equipment, material and services was provided by suppliers based on the market conditions and expectations applicable at the time of developing the estimate.

The market conditions are susceptible to the impact of demand and availability at the time of purchase and could result in variations in the supply conditions. The estimate in this Report was based on information provided by suppliers and assumes there are no problems associated with the supply and availability of equipment and services during the execution phase.

21.1.4 Capital Cost Estimate

21.1.4.1 Summary

The process estimate was derived from data as shown on flowsheets and general arrangement drawings, models, MTOs and lists and includes all associated infrastructure. The estimated pre-production capital costs for design, construction, installation, and commissioning for all the facilities and equipment are shown by major discipline in Table 21-2.

Table 21-2 Process Plant

DISC. CODE	WBS DESCRIPTION	INITIAL CAPEX ($M CAD)
4010	ARCHITECTURAL	37.3
1010	EARTHWORKS	19.7
2010	CONCRETE	19.5
3010	STRUCTURAL STEEL	9.4
5055	PLATEWORK	15.9
5010	MECHANICAL EQUIPMENT	73.3
6010	PIPING	14.9
7010	ELECTRICAL EQUIPMENT	12.5
7010	ELECTRICAL BULKS	10.1
8010	INSTRUMENTATION	6.2
5080	MOBILE EQUIPMENT	4.7
	THIRD PARTIES	2.1
	TOTAL DIRECT COSTS	**225.6**
9610	PROJECT DELIVERY	39.6
9010	FIELD INDIRECTS	27.4
9020/9030/9070	SPARES / FIRST FILLS / VENDOR REPS	7.7
9040	FREIGHT AND LOGISTICS	8.9
	TOTAL INDIRECT COSTS	**83.6**
9910	PROVISIONS / CONTINGENCY	**46.4**
PROJECT TOTAL		**355.6**

21.1.5 General Methodology

The capex was a quantitative based cost estimate, with engineering developed material take-offs with factored quantities, semi-detailed unit costs, and budgetary quotations for major equipment.

The structure of the estimate was a build-up of the direct & indirect cost of the current quantities; this includes the installation/construction hours, unit labour rates and construction equipment costs, bulk and miscellaneous material and equipment costs, any subcontractor costs and freight.

The methodology applied to develop the estimate was as follows:

- Define the scope of work;
- Quantify the work in accordance with standard commodities;
- Organize the estimate structure in accordance with agreed Work Breakdown Structure;
- Develop a priced Mechanical Equipment List and Electrical Equipment List;
- Determine bulk material pricing;
- Determine the installation cost for equipment and bulks;
- Establish requirements for freight;
- Determine and agreed on foreign exchange rates;
- Determine/develop indirect costs;
- Determine the estimate contingency value;
- Undertook internal peer reviews, finalize the estimate, and obtain sign off by the Project Manager and Qualified Professional.

21.1.6 Exchange Rates

The exchange rates in Table 21-3 were used to develop the capital cost estimate for the PFS.

Table 21-3 Estimate Exchange Rates

Code	Currency	Exchange Rate
C$	Canadian Dollar	1.00 C$ = 1.00 C$
US$	US Dollar	1.00 C$ = 0.763 US$
AUD	Australian Dollar	1.00 C$ = 1.045 AUD
EUR	Euro	1.00 C$ = 0.643 EUR

21.1.7 Overall Site

The estimated overall site costs of $17M include for the bulk earthworks for plant, stockpile, primary and pebble crushers, ROM pad, infrastructure pads, and plant site roads.

21.1.8 On-Site Infrastructure Capital Costs

The On-site infrastructure initial capital costs of $41.6M consist of bulk fuel storage and distribution, plant site ancillary buildings including truck shop complete with warehouse and various maintenance shops, a combined administration and mine dry building, cold storage warehouse, assay laboratory, sewage collection and treatment, 265 bed refurbished construction camp, which is not utilized during operations. The 50 existing exploration camp beds will be used for construction needs, bringing the total available beds for construction to 315. Camp housekeeping and catering during the pre-production is also included for 315 persons. Additionally, raw water supply, an incinerator for waste management, construction laydown area and an allowance to temporarily relocate overhead power line to allow over height equipment to pass during delivery to site.

21.1.9 Off-Site Infrastructure Capital Costs

21.1.9.1.1 Access Road and rerouting of Spanish Lake Road

The off-site infrastructure initial capital costs of $7.8M consist of a site access road and rerouting of Spanish Lake Road to avoid a proposed waste rock and the impact of the ultimate pit.

21.1.9.1.2 Off-site Electrical

The off-site electrical initial capital costs of $83.1M consisted of overall site electrical (including the 138kV receiving substation), 138kV Main Substation (BCHydro), a 138kV overland transmission line with the associated service roads and fibre optic. Additional sustaining costs of $5.8M are also included in the capital estimate.

21.1.10 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:

- Construction Indirects:
- Spares;
- First Fills;
- Freight and Logistics;
- Commissioning and Start-up;
- EPCM and Expenses;
- Vendors.

21.1.10.1 Construction Field Indirects

Construction Field indirects are items or services, which are not directly attributable to the construction of specific physical facilities of plant or associated infrastructure but are required for support during the construction period.

The construction contractor's indirects (distributable costs) were allocated by discipline at a $56.07 per direct manhour rate (except for concrete works, which is $7.43/h), based on a local contractor's returned data and are included at $27.4M.

21.1.10.2 Spares

Costs for capital, operating and commissioning spares were factored and equate to $4.22M.

21.1.10.3 First Fills

Costs for first fills were factored as $1.92M.

21.1.10.4 Freight and Logistics

Freight costs were deemed to include inland transportation, export packing, all forwarder costs, ocean freight (where applicable) and insurance, receiving port custom agent fees, and local inland freight to the project site.

Freight was calculated on each line item in the estimate (where applicable) as a percentage of the material and/or equipment cost and summed by each discipline in the Indirect costs. Sub-Contract items are deemed to include freight; therefore, no additional freight has been included on these items.

Costs for freight and logistics were estimated at $8.93M.

21.1.10.5 Commissioning and Start-up

Commissioning assistance was factored as $1.6M.

21.1.10.6 EPCM

EPCM services costs cover such items as engineering and procurement services (home office based), construction management services (site based), project office facilities, IT, staff transfer expenses, secondary consultants, field inspection and expediting, commissioning, corporate overhead, and fees.

The EPCM costs were calculated at approximately $38M for Ausenco’s scope of work.

21.1.10.7 Vendor Support

Costs for vendor representatives for construction and commissioning were factored as $1.54M.

21.1.10.8 Escalation

No escalation beyond the base date of Q1-2021 was proportioned to any part of the estimate.

21.1.10.9 Contingency

Estimate contingency was included to address anticipated variances between the specific items contained in the estimate and the final actual Project cost.

The amount of risk was assessed with due consideration of the level of design work, methods by which way pricing was derived, and the preliminary nature of the Project implementation plan.

The estimate contingency was allocated as $46.4M.

21.1.11 Tailings Storage Facility and Water Management

21.1.11.1 Initial Direct and Sustaining Capital Costs

An initial and sustaining capital costs estimate was completed for the components of waste and water management presented on Figure 18-1 and outlined below. A detailed cost estimate is provided in the PFS Tailings and Water Management Report (KP, 2021) with the Basis of Estimate included in Appendix C-5.

- Site preparation for the TSF embankment and water management structures, which includes, where applicable:
    - Clearing and grubbing of the footprints (TSF embankment and water management structures),
    - Sediment and erosion control, construction dewatering and seepage controls using BC Ministry of Environment Best Management Practices (BMPs),
    - Excavation and removal of topsoil and unsuitable overburden materials.
- Access road construction for the following:
    - The TSF West access road to Boswell Lake,
    - The South Seepage Collection Pond access road,
    - The North Seepage Collection Pond access road,
    - The TSF east diversion ditch access road,
    - Access roads to the TSF borrow areas (assumed to be located within the TSF basin),
    - The Water Management Pond access road, and
    - The access road from the Water Management Pond to the Plant Site.
- Earthworks costs for both TSF embankments, including initial cofferdam construction.
    - Earthworks costs are integrated between mining costs, the tailings and water management costs, with mining costs covering material haulage from the Open Pit to the TSF Embankments.
- TSF embankment monitoring instrumentation (piezometers and inclinometers).
- Water management pond construction, including liner installation
- Diversions ditches, Boswell Lake diversion embankment, and the south diversion channel.
- Sediment control ditches for waste rock stockpiles.
- Tailings distribution, reclaim, and seepage collection and recycle systems which include associated pumps, pipelines, fittings, and other components as outlined in the detailed KP cost estimate. (PFS Tailings and Water Management Report (KP, 2021) with the Basis of Estimate included in Appendix C-5).
- Revegetation for the progressive reclamation of the Waste Rock Storage Facilities (WRSF’s).

21.1.11.2 Indirect Costs

The indirect costs include the following:

- Contractor mobilization and demobilization.
- A contractor travel allowance assuming 3:1 week rotation. Annual person estimated are based on the number of personnel over the assumed construction periods of each year.
- On site engineering construction management and oversight
- Construction of an engineering site lab
- Site investigation programs
- Engineering Procurement.

The cost estimate is compiled using engineering experience and unit rates built up using first principles based on standard contractor rates. Equipment fleet and unit rates were obtained using the BC Blue Book (BC Hydro, 2019) and/or RS Means (RS Means, 2020) rates. The detailed cost estimate is provided in the PFS Tailings and Water Management Report (KP, 2021).

A summary of the capital and sustaining costs relating to the tailings and water management systems identified above is provided in Table 21-4. This summary does not include any mining costs covering select material haulage costs from the Open Pit.

Table 21-4 Tailings and Water Management Capital and Sustaining Costs

Description	Initial Capital ($'000s)	Sustaining Capital ($'000s)
Ore Handling	-	2,684
TSF Access Roads	196	-
Tailings Disposal	19,992	29,472
Tailings Reclaim	5,104	146
Water Management	15,026	3,309
Sub-Total	40,318	35,611
Indirects	8,364	12,849
Contingency	9,163	8,233
TOTAL	57,845	56,693

21.1.11.3 Contingency

21.1.11.3.1 Tailing Storage Facility and Water Management

The tailings and water management cost estimates includes a contingency on the direct costs of 25 % (earthworks) and 20 % (mechanical systems). These contingencies are to account for "known unknowns" conditions related to the items within the project scope.

21.1.12 Environmental

All permitting acquisition activities and any associated physical works (e.g., fish habitat compensation construction) are assumed to be sunk costs incurred by the Company prior to the construction start (Year

-2). However, because of permit requirements, it is expected that a range of monitoring and reporting activities will be realized annually starting in the first year of construction (Year -2). These include:

- Surface water quality monitoring
- Surface flow monitoring
- Ground water quality monitoring
- Hydrogeology monitoring
- Fisheries and aquatic monitoring
- Wildlife monitoring
- Air Quality Monitoring
- Noise Monitoring
- Other miscellaneous monitoring of permit conditions.

A total of $8.8 M has been allocated for environmental monitoring and reporting during construction and operations. A summary of the environmental costs per period is provided in Table 21-5.

Table 21-5 Environmental Costs

Description	Initial Capital	Sustaining Capital
	($'000)	($'000)
Environmental	1,600	7,200

21.1.13 Water Treatment Facilities

The Water Treatment was provided in two phases, Active and Passive Treatment. One location, the Water Treatment Plant processed the Active Treatment and four other locations handled the Passive Treatment on-site (North RSF, west RSF, North Seepage Collection Pond, and the Pit Lake location.

The active and passive treatment cost estimates were developed using capacity factors, parametric models, professional judgment, and analogy and are considered Class 5. The Class 5 accuracy level is considered consistent with the current level of development of the water quality and water balance models for the Project. Costs were developed in the first quarter of 2021 and are not discounted or inflated for future years; for items sourced in the United States, a US$:C$ exchange rate of 1.25 was utilized.

A total initial capital of $17.9M and $48.2M sustaining capital was allocated for water treatment on-site.

21.1.14 Sustaining Capital Cost Estimate

Sustaining Capital is defined as the annual capital required for replacing plant equipment and components that have served their useful life. Equipment upgrades, annual maintenance or operating expenses are not included within these cost estimates and are treated separately.

The LOM sustaining capital cost estimate was broken into the following areas as summarized in Table 21-1:

- Mining
- Processing and Infrastructure
- Tailings and Water Management
- Environmental
- Water treatment

Refer to Table 21-1 for Sustaining Capital costs.

21.2 Closure Cost Estimate

The following Table 21-6 presents the Closure Costs for the project. Mill, TSF, water management, water treatment and infrastructure closure estimates have been prepared as of the date of this report. The closure cost for the SMG is $159.6 million. Details of the Closure Costs are presented in Appendix C-1.

Table 21-6 Closure Costs

Description	Closure Cost ($ '000s)
Direct Closure Costs	
Mining	32,127
Process and on-site infrastructure	14,690
TSF and Water Management	62,222
Water Treatment	24,958
Environmental (including passive closure)	2,600
Directs Subtotal	**136,597**
Project Closure In-direct Costs	
Owner's Costs	4,790
Indirect Costs	2,593
Contingency	
Contingency	15,622
Total Closure Costs	**159,602**

21.2.1 Mining (Reclamation)

Closure costs related to the stockpiles, haul roads and open pits are prepared by MMTS. Estimated closure costs include the earthworks related to the following activities:

- Re-grading stockpiles to overall slope angles.
- Re-grading of selected areas of the open pit highwalls (~35% of total exposed areas).
- Scarification of haul roads.
- Placement of 0.3 m of overburden on the tops and sides of the stockpiles and selected areas of the open pit.

- Placement of 0.1 m of overburden on the haul roads.
- Placement of 0.3 m of topsoil on the tops and sides of stockpiles and selected areas of the open pits
- Placement of 0.1 m of topsoil on the haul roads

Estimated closure costs unit rates are based on the using of the owner operated mine fleet for completing the activities listed above. Costs for revegetation are included elsewhere in the project. Progressive reclamation is assumed, with closure costs applied when areas are assumed available for reclamation.

21.2.2 Process and Infrastructure

The process and infrastructure closure cost is estimated at $14.7M for the process plant and associated infrastructure. These costs include the following:

- Removal of all machinery, equipment and building structures.
- Concrete foundations covered and revegetated.
- Disposal of scrap material.
- Disposal of chemical and reagents.

21.2.3 TSF and Water Management – Reclamation and Salvage Values

The Progressive reclamation is planned between Year 7 and Year 14 for the WRSF's. The reclamation of these items is included in the operations period (sustaining).

The tailings and water management systems require a variety of closure and reclamation activities during the active closure period, as outlined and presented in the PFS Tailings and Water Management Report (KP, 2021). This includes:

- TSF embankment reclamation,
- North embankment spillway construction,
- Tailings surface capping,
- Construction of PTSs,
- Decommissioning and removal of the WMP and select water management structures.

A summary of the closure costs relating to the tailings and water management systems identified above is provided in Table 21-17. This summary does not include any mining costs covering select material haulage costs from the Open Pit or construction and maintenance of the passive water treatment systems.

Table 21-7 Tailing and Water Management Closure Cost Estimates

Description	Active Closure (Y15 –Y18)	Passive Closure (Y19-Y21)
	($'000s)	($'000s)
Tailings Disposal	58,721	-
Water Management	3,501	-
Sub-Total	62,222	-
Indirects	2,593	-
Contingency	15,622	-
TOTAL	80,437	-

21.2.4 Environmental Monitoring During Closure

A total of $2.6M has been allocated for environmental monitoring and reporting during active and passive closure periods. A summary of the environmental costs during closure per period is provided in Table 21-8.

Table 21-8 Environmental Closure Costs

Description	Active Closure (Y15-Y18)	Passive Closure (Y19-Y21)
	($'000)	($'000)
Environmental	1,600	1,000

21.2.5 Water Treatment Facilities

A total estimated cost of $25M has been allocated for closure of the Water Treatment Plant and the Passive Water Treatment facilities on site.

21.3 Operating Cost Estimate

The estimated Life of Mine annual operating costs are presented in Table 21-9.

Table 21-9 Unit Operating Cost LOM

Area	Unit Cost $/t Milled
Mining	10.80
Process and G&A	7.67
Water Treatment	0.47
Tailings and Water Management	0.17
Owners G&A	0.27
Total ($/t milled)	19.38

21.3.1 Mining Operating Costs

Mine operating costs are built up from first principles. Inputs are derived from vendor quotations and historical data collected by Moose Mountain Technical Services. This includes quoted cost and consumption rates for such inputs as fuel, lubes, explosives, tires, undercarriage, GET, drill bits/rods/strings, machine parts, machine major components, and operating and maintenance labour ratios. Labour rates for planned hourly and salaried personnel have been supplied by Spanish Mountain Gold.

Estimated annual and life-of-mine unit mining costs are shown Table 21-10.

Table 21-10 Unit Mine Operating Costs, $/t mined & $/t milled

Operation	$/t Mined	$/t Milled
GRADE CONTROL	$0.09	$0.43
DRILLING	$0.21	$1.04
BLASTING	$0.27	$1.31
LOADING	$0.29	$1.40
HAULING	$0.80	$3.87
PIT SUPPORT	$0.33	$1.60
SITE DEVELOPMENT	$0.07	$0.34
DIRECT COSTS - Subtotals	**$2.06**	**$10.00**
Mine Operation/Maintenance GME	$0.11	$0.52
Mine Engineering GME	$0.06	$0.28
TOTAL GME COSTS	**$0.16**	**$0.80**
TOTAL OPERATING COST	**$2.22**	**$10.80**

21.3.2 Mine Operations

The mine will operate 365 days per year, 24 hours per day with two 12-hour shifts per day. Four shifts are specified, all one week on and one week off: one crew on dayshift, one crew on night shift, and two crews off, drive-in and drive-out. An allowance of 5 days of no production has been built into the mine schedule to allow for adverse weather conditions.

21.3.3 Mine Production Schedule

Annual ore production tonnes, waste tonnes, and stockpiled management tonnes, are taken from the PFS mine production schedule. Drilling, loading, and hauling hours are calculated based on the capacities and parameters of the equipment fleet. These tonnes and hours also provide the basis for blasting consumables, and support fleet inputs.

21.3.4 Grade Control Drilling Inputs

Grade control drilling is applied to all scheduled mineralized material (ore and waste) to "look ahead" at upcoming benches and better define mineralization boundaries for controlled blasting and loading operations. A requirement for Reverse Circulation (RC) grade control drilling hours is calculated with inputs from hole size, pattern dimensions, bench height, material density, and penetration rate of the drill.

Additional costs are added to the grade control drill for sampling and assaying on 2 m intervals. Costs for assay lab technicians are also included.

21.3.5 Production Drilling Inputs

Based on the tonnes scheduled, a requirement for production drilling hours is calculated with inputs from hole size, pattern dimensions, bench height, material density, and penetration rate of the drill.

Drilled patterns and depths are applied in estimated wet and dry mining zones. Patterns and collars are defined to target specific powder factors.

Trim blasting in 10% of mined rock is planned to be drilled on an alternative pattern and depth.

No drilling is assumed in topsoil and overburden materials.

21.3.6 Blasting Inputs

Powder factor of 0.26 kg/t (0.70kg/BCM) are estimated for the ore and waste rock. The quantity of explosives is calculated and costed for the targeted powder factor, pattern area, and explosive density. In addition, an estimate for initiation systems and blasting accessories are made on a per hole basis which includes detonation cord, a booster, and electric detonator. As an emulsion base product is assumed, no liners are included in the per hole pricing.

Explosive Blasting operations are planned as a supplier operated function. Additional costs are included for delivery of the product to the site, site storage, delivery of the product to the hole, hole loading and shooting blasts, as well as coverage for lease costs for necessary equipment and facilities (pickup trucks, blasting trucks, stemming loader, storage facilities, magazine, garage, trailers, and fencing).

21.3.7 Loading & Hauling Inputs

Fleet requirements for loading and hauling are calculated based on loader and hauler productivities applied to the mine production schedule.

Loader productivities for various loading conditions are estimated and applied to the scheduled material movement to produce required equipment operating hours. The wheel loader is also planned to load the primary crusher for 25% of the mill feed tonnages.

Haulage profiles are estimated from pit centroids at each bench to designated dumping points for each scheduled period. These haul profiles are inputs to a haul cycle simulation program and the resulting cycle times are used to estimate required hauler operating hours and fuel burn in each scheduled period.

Annual average hauler productivities for the 140-tonne payload haulers range from 320 t/h to 500 t/h. Stockpile reclaim productivities are assumed to be 680 t/h.

Articulated haulers are assigned to topsoil stripping activities as well as operating hours initially assigned to the 140-tonne payload hauler fleet at a ratio of 34.2/128.1.

All productivities listed above are on a NOH (net operating hour) basis.

21.3.8 Pit Support Inputs

Pit services include:

- Haul road development and maintenance
- Pit floor and ramp maintenance
- Stockpile maintenance
- Ditching
- Progressive Reclamation
- Mobile Fleet fuel and lube support
- Open pit dewatering
- Topsoil excavation
- Secondary blasting and rock breaking
- Open pit lighting
- Mine safety and rescue
- In pit transportation of personnel and operating supplies
- Snow Removal

A fleet of mobile equipment is specified to handle these pit support activities. Annual utilization of this support equipment is driven by the utilization of the primary equipment in the fleet.

21.3.9 Equipment Operating Cost

All equipment is costed using quoted or estimated fuel consumption rates, consumables costs, GET estimates, labour ratios and general parts and preventive maintenance costs, per hour, or per hour interval. The hourly rates are then multiplied by the operating hours of the machine to find a constant distributed operating cost per operating or working hour.

The costs for major components of the larger equipment types are calculated separately from the distributed hourly cost. Major repairs are clocked with the usage of the piece of equipment so that major repairs costs are represented in the year it occurs, rather than averaging this cost over many years. This method gives a more representative cash flow. Equipment replacement is clocked in the same manner, so that individual equipment units cumulative operating hours are tracked up to a set limit, and then a replacement is introduced, and sustaining capital costs incurred in that year.

Running hours (Service Metre Unit) on each piece of equipment are estimated based on operating capacities and requirements of the mine production schedule. These Service Metre Unit hours are multiplied by the hourly consumables rates and unit operating costs to calculate the total equipment operating costs for each year of operation.

Diesel price of $0.92/L is used. This value includes provision for delivery and storage on site, as well as applicable provincial and federal taxes.

21.3.10 Hourly Labour

Labour workhour ratios are categorized for the different labour types (e.g., operators, mechanics, electricians, etc.) and assigned to each piece of equipment, and then multiplied by the operating hours. The total hours required for each category are added together and rounded off to assign a full person to each crew; any additional hours remaining, after rounding, are grouped together into an unallocated labour pool. Table 21-11 shows a summary of mine hourly labour counts.

21.3.11 Mine GME

GME is a category for mine operation’s overhead and technical services costs. It consists of costs for all salaried staff, a consumable and rental allowance, staff travel allowance, and software and fleet management and engineering systems’ licensing and maintenance. This category is a fixed cost, and does not vary by production or fleet size, except for ramp-ups to full staffing and ramp-downs at the end of mine life. Table 21-12 shows a summary of estimated salaried staff and technical personnel.

21.3.12 Mine Operations Site Development Costs

Mine operations site development costs are described below:

- Clearing & Grubbing – The costs for clearing and grubbing are estimated for the pits, haul road, and stockpile areas. The costs are incurred prior to those areas being required for mine operations. Any cost discounts for not recovering non-merchantable timber have not been accounted for.
- Geotechnical Drilling – A program of dewatering and depressurization drilling is estimated for the open pit. It involves the drilling and installation of vertical dewatering wells with associated pumping, as drilling of horizontal drain holes in the pit highwall for passive dewatering. Vertical Well installation costs are estimated based on 400 m depths and includes all collaring and

installation of required pumps. Horizontal drilling costs are based hole requirements of 1 m for every 1 m of highwall exposure.

- Topsoil Excavation – Topsoil quantities for pit stripping are included in the mine production schedule, with loading and hauling hours accounted for in the mine fleet. Additional topsoil stripping quantities for the haul roads and stockpile footprints are also estimated. Topsoil hauling productivities of 100 m^3/hour are based on 1.5 km hauling distances for the articulated haulers. Hydraulic excavator topsoil excavation productivity is estimated to be two times the hauler productivity.
- Crusher Rock Production – An estimate to produce crush rock for mine operations is included. Crush rock will be used for haul road maintenance, as well as for stemming materials in blasting. Haul roads assume 0.5 m crush rock topping when constructed and 0.1 m of each road are resurfaced per year. Stemming quantities are estimated based on blastholes produced per year and stemming length in each blasthole. The costs to supply this material assume that mined waste from the pit will be run through a contractor crusher.

Table 21-11 Mine Hourly Labour Summary

Position	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14
MINE OPERATIONS															
Drill Operator	8	12	12	12	12	12	16	16	16	16	12	12	8	4	0
Blasters / Mixers / Helpers	8	10	10	10	10	12	12	12	12	12	12	12	10	6	0
Excavator / Loader Operator	8	12	12	12	12	12	16	16	16	16	16	16	8	8	4
Haul Truck Driver	40	46	46	44	48	52	72	72	72	72	72	52	32	20	8
Grader Operator	8	8	8	8	8	8	12	12	12	12	12	8	8	4	4
Track Dozer Operator	12	16	16	16	16	16	24	24	24	24	20	20	12	8	4
Water/Fuel Truck Operator	8	8	8	8	8	8	12	12	12	12	12	8	8	4	4
MINE MAINTENANCE															
Electrician	4	6	6	6	6	6	8	8	8	8	6	6	4	2	2
HD Mechanic	14	20	20	20	20	20	28	28	28	28	26	22	14	8	3
LD Mechanic	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Machinist/Welder	6	10	10	10	10	10	14	14	14	14	12	10	6	4	2
Labourer	4	6	6	6	6	6	8	8	8	8	8	6	4	2	2
TOTAL PERSONS	**122**	**156**	**156**	**154**	**158**	**164**	**224**	**224**	**224**	**224**	**210**	**174**	**116**	**72**	**34**

Table 21-12 Mine Salaried Staff Summary

Position	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14
MINE OPERATIONS															
Mine Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine General Foreman	0	1	1	1	1	1	1	1	1	1	1	1	1	0	0
Pit Shift Foreman	4	4	4	4	4	4	4	4	4	4	4	4	4	4	2
Drill and Blast Foreman	1	2	2	2	2	2	2	2	2	2	2	2	2	1	0
Clerks	1	2	2	2	2	2	2	2	2	2	2	2	2	1	0
Safety/Training Officer	3	3	2	2	2	2	2	2	2	2	2	2	1	1	0
Pit Labourer	4	8	8	8	8	8	8	8	8	8	8	8	8	4	0
Pump Crew	0	4	4	4	4	4	4	4	4	4	4	4	4	4	0
Dispatch Controllers	0	4	4	4	4	4	4	4	4	4	4	4	4	0	0
Assay Lab Technicians	2	4	4	4	4	4	4	4	4	4	4	4	4	4	0
MINE MAINTENACE															
Maintenance General Foreman	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Maintenance Shift Foreman	0	2	2	2	2	2	2	2	2	2	2	2	2	0	0
Maintenance Planner	0	1	1	1	1	1	1	1	1	1	1	1	0	0	0
Maintenance Clerk	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0

Position	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14
TECHNAL SERVICES															
Technical Services Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0
Senior Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0
Geologists	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Ore Grade Technicians	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Senior Mining Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0
Mine Engineer	1	2	2	2	2	2	2	2	2	2	2	2	2	0	0
Dispatch Engineer	0	1	1	1	1	1	1	1	1	1	1	1	0	0	0
Mine Technicians	1	2	2	2	2	2	2	2	2	2	2	2	2	0	0
Surveyor	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Geotechnical Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
TOTAL PERSONS	**28**	**52**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**51**	**48**	**26**	**6**

21.3.13 Processing Plant Operating Costs

Process operating costs are built-up from first principles. Table 21-13 summarizes the annual process operating cost breakdown for the 7.3Mt per year.

Table 21-13 Process Plant Unit Operating Cost

Area	Unit Cost ($/t milled)
Labour	1.29
Power	1.73
Operating Consumables and Reagents Costs	3.22
Plant Maintenance	0.25
Mobile Equipment	0.04
Total ($/t milled)	6.53

21.3.13.1 Power

Power costs were calculated from an estimate of annual power consumption and using a unit cost of $0.065/kWh.

Power consumption was derived from the calculated power draw of equipment listed in the mechanical equipment list. The average on-line process plant power draw was estimated at 24.7MW.

Annual process plant energy consumption was estimated at 194,201 MWh, for a cost of $12.6M.

21.3.13.2 Operating Consumables and Reagents Costs

Processing reagent and consumable costs were estimated based on the throughput and testwork supported consumption rates. Reagent costs for 7.3Mt per year are summarised in Table 21-14.

Table 21-14 Reagent Costs

Area	Annual Cost (C$M)
Collector (PAX)	3.35
Frother	1.92
Hydrated lime	0.55
NaCN	2.83
Oxygen	0.23
Activated carbon	0.03
NaOH	0.51
HCl	0.10
SMBS	1.13
$CuSO_4$	0.11

Area	Annual Cost (C$M)
CMC	1.16
Total	**11.92**

Reagent costs were based on:

- Consumption rates determined during testwork;
- Data base unit costs for the reagents including freight.

Costs for liners were estimated based on vendor information and benchmarking similar plants. Costs for mill balls were estimated for expected consumption based on an abrasion index (Ai) of 0.21. These costs for 7.3Mt per year processing are summarized in Table 21-15.

Table 21-15 Unit Operating Costs, SAG and Ball Mills

Area	Annual Cost C$M
SAG Mill Balls	4.05
Ball Mill Balls	3.09
Regrind Media	0.06
SAG Mill Liners	1.37
Ball Mill Liners	0.90
Regrind Liners	0.14
Total	9.61

21.3.13.3 Plant Maintenance

Annual maintenance spares and consumable costs were estimated at 5% of the mechanical equipment costs.

This results in an annual maintenance consumable cost estimate of $1.84 M.

21.3.13.4 Labour

Labour costs include all processing and maintenance costs.

Costs were estimated from a breakdown of staffing positions, estimated at 85 in total, excluding G&A manpower.

The annual processing labour cost was estimated at $9.45 M inclusive of all applicable burdens. Labour costs are presented in Table 21-16.

Table 21-16 Labour Costs

Grade	Job title	Rotation Schedule	No. Employees
Mill Administration			
	Mill Manager	14 On/14 Off	1
Mill Operations			
	Operations Superintendent	14 On/14 Off	1
	Operations Shift Supervisors	14 On/14 Off	3
	Control Room Operator	21 On/7 Off	6
	Primary Crusher Operator	21 On/7 Off	4
	Grinding Operator	21 On/7 Off	4
	Flotation Operator	21 On/7 Off	4
	Leach Operator	21 On/7 Off	3
	Gold Room Operator	21 On/7 Off	3
	Tailings Operator	21 On/7 Off	4
	Labours / helper	21 On/7 Off	8
Mill Maintenance			
	Maintenance Superintendent	21 On/7 Off	1
	Maintenance Supervisors	21 On/7 Off	2
	Maintenance Planner	21 On/7 Off	1
	Electrical Planner	21 On/7 Off	1
	Elect/Instr. Superintendent	21 On/7 Off	1
	Electrical Supervisors	21 On/7 Off	3
	Millwrights	21 On/7 Off	6
	Welders	21 On/7 Off	4
	Pipe Fitters	21 On/7 Off	2
	Instrumentation Tech	21 On/7 Off	2
	Electrician	21 On/7 Off	6
Mill Metallurgy			
	Chief Metallurgist	14 On/14 Off	1
	Senior Metallurgists	14 On/14 Off	1
	Junior Metallurgists	14 On/14 Off	3
Mill Assay Laboratory			
	Chief Assayer	14 On/14 Off	1
	Sample Preparation	21 On/7 Off	4
	Assayer (Wet and Fire)	14 On/14 Off	5
	Total		**85**

21.3.13.5 Mobile Equipment

Annual plant site mobile equipment costs were estimated at $0.29M.

21.3.14 General and Administrative Costs

The total annual cost for G&A items is estimated to be $7.87M, or $1.08/t milled.

G&A costs included all salaried and hourly labour not assigned to mine or process operations, such as:

- General Management;
- Administration including camp administration;
- Human Resources;
- Reception;
- Health, Safety and Environmental;
- Security;
- Procurement and warehousing;
- Accounting;
- IT;
- Janitorial;
- Community Relations and environmental.

It also included consumables and contractors not covered under the mine and process operations, such as:

- Office supplies and stationery;
- Professional associations and publications;
- Insurance;
- Travel;
- Site communications;
- Computer and IT services;
- Site services;
- Corporate office expenses;
- Recruitment;
- Training;
- Infrastructure power;
- Protective equipment and training supplies;
- Medical Services and First Aid Supplies;
- Camp meals;
- Site transport;
- Off-site transport;
- Propane fuel;

21.3.15 Tailings Operating Costs

Operating costs for the tailings and water management systems have been estimated to cover the following systems, as outlined, and presented in the PFS Tailings and Water Management Report (KP, 2021):

- Rougher tailings distribution system,
- Cleaner tailings distribution system,
- TSF reclaim systems (Main TSF and PAG Cell)
- North seepage collection pond system,
- South seepage collection pond system,
- Boswell Lake diversion system,
- Water Management Pond reclaim system,
- Water Treatment Plant source reclaim system,
- North WRSF seepage collection system, and
- Open pit dewatering system.

The tailings systems are operational through the mine life. The tailings reclaim and North and South seepage collection systems continue to operate through the active closure period (Y15-Y18) before being decommissioned. The Boswell Lake diversion system will operate through Year 10, at which point the south diversion channel is constructed and the pumping system will be decommissioned. The WMP and WTP systems are operational through operations and active closure, while the open pit dewatering systems is decommissioned after Year 13 of operations.

A summary of the tailings and water management operating costs is presented in Table 21-17 below.

Table 21-17 Tailings and Water Management Operating Costs

Description	Operations (Y1-Y14) (C$)	Active Closure (Y15-Y18) (C$)	Passive Closure (Y19-Y21) (C$)
Tailings Disposal	$ 7,054,700	$ 0	$ 0
Tailings Reclaim	$ 3,995,300	$ 231,000	$ 0
Water Management	$ 4,818,000	$ 863,500	$ 0
TOTAL	**$ 15,868,000**	**$ 1,094,500**	**$ 0**

21.3.16 Water Treatment Facilities

Operating costs for the Active Treatment systems for the Water Treatment Plant include costs for power, labor, replacement parts, and chemicals. Periodic costs to replace major equipment in the WTP are not anticipated as the duration of plant operations (less than 20 years) is shorter than the lifespan of this type of equipment. Active operational costs will decrease in active closure as treatment is transitioned to passive.

PTSs are basically operator-free but not maintenance-free. Annual costs will include inspection and maintenance of the civil infrastructure, berms, conveyance, liners, vegetation, etc. Periodic costs (every 20 years) include media replacement along with the liners, geotextile, piping, and drainage gravel for BCRs. For the iron terrace, annual spreading of iron hydroxide will be required. Wetlands maintenance would be included with the annual infrastructure maintenance effort.

The total LOM operational Water Treatment costs was estimated at $45.2M.

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 several known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Information that is forward-looking includes:

- Mineral Resource and Mineral Reserve estimates
- Assumed commodity prices and exchange rates
- Mine production plans
- Projected recovery rates
- Sustaining and operating cost estimates
- Assumptions as to closure costs and closure requirements
- Assumptions as to 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 mineralised material, grade, or recovery rates
- Geotechnical and hydrogeological considerations during mining being different from what was assumed
- Failure of plant, equipment, or processes to operate as anticipated
- Accidents, labour disputes and other risks of the mining industry.

22.2 Economic Analysis

All dollar amounts in this analysis are expressed in Q1 2021 Canadian dollars, unless otherwise specified.

The economic analysis is run over the entire project life, comprising two years of construction and 14 years of mining and milling. The valuation date on which the Net Present Value (NPV) and Internal Rate of Return (IRR) are measured is the commencement of construction in Year -2. Corporate sunk costs to that point in time, including costs for exploration, technical studies, royalty buyback, and permitting, are not included in cash flow; except when estimating tax. The project cashflow assumes 100% equity financing.

Spanish Mountain Gold's taxation model, as of Q2 2021, has been used to estimate federal, provincial, and other taxes. Some additional details are included in Section 22.3.

The basis of the project economic analysis is summarized in Table 22-1. Details of the capital and operating cost estimates are described in Section 21.

Table 22-1 Inputs for Economic Analysis

Parameter	Value	Units
Gold Price	$1,600	US$/oz
Silver Price	$24	US$/oz
Currency Exchange Rate	0.76	C$:US$
Gold Payable	99.8%	
Silver Payable	90.0%	
Gold Refining Terms	$1.00	$/oz
Silver Refining Terms	$0.60	$/oz
Doré Transport Costs	$1.00	$/oz
Doré Insurance Costs*	0.15%	
Royalty**	1.5%	
Gold Process Recovery	90%	
Silver Process Recovery	40%	
Mining Cost***	$2.22	$/t mined
Mining Cost	$10.80	$/t milled
Processing Costs	$6.58	$/t milled
General & Administration Costs	$1.36	$/t milled
TMF Operating Costs	$0.17	$/t milled
Water Treatment Operating Costs	$0.47	$/t milled
Total Operating Costs	$19.38	$/t milled

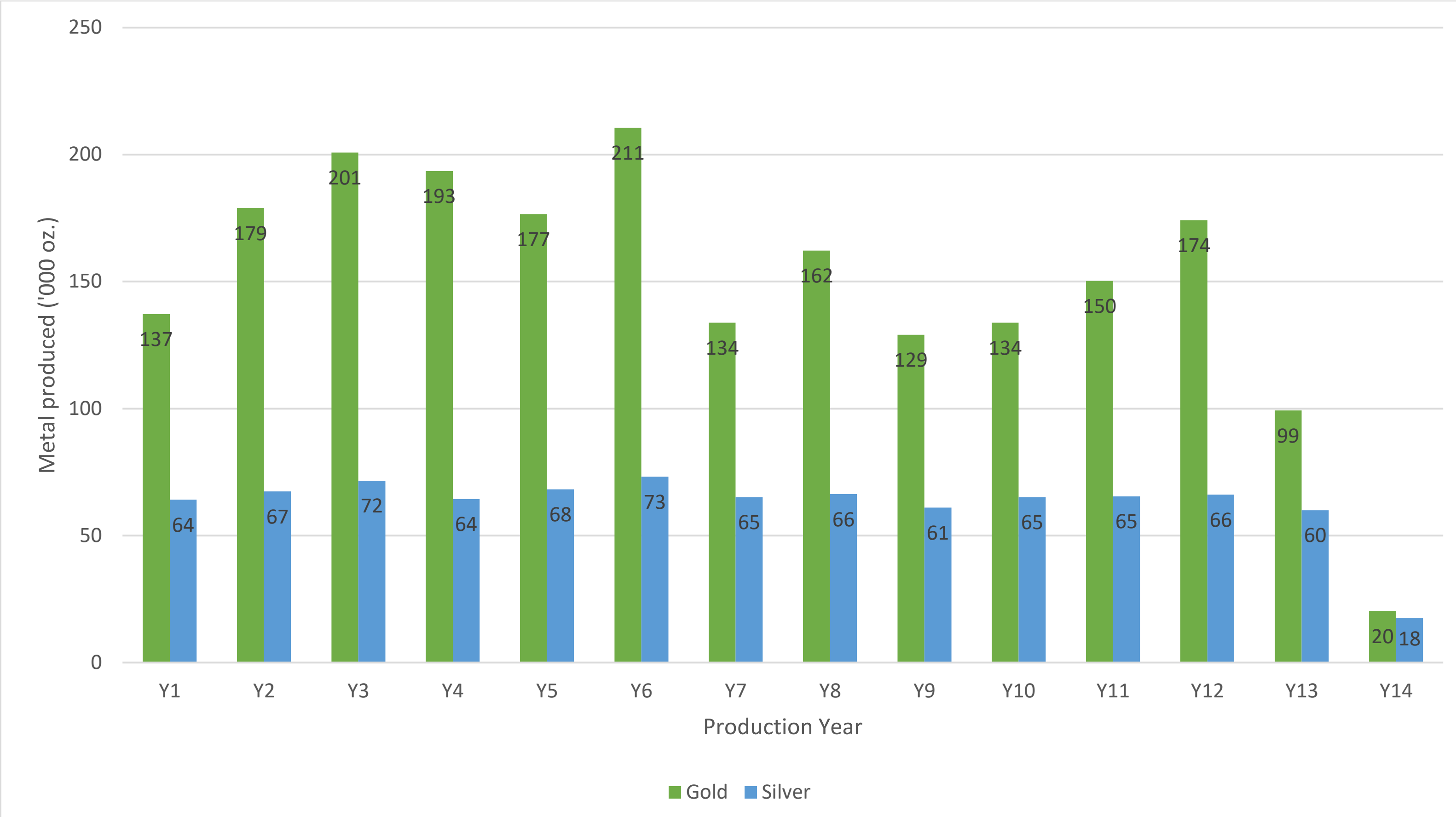
** % of Net Value after smelter charges have been applied.*
*** It is anticipated that NSR obligations under the 'Wallster and McMillan' and 'R.E. Mickle' claims, described in Section 4, will be purchased by the owner in advance of commercial production, lowering the overall NSR commitment within the delineated resource to 1.5%. The cost of this purchase has not been included in the project cashflow.*
**** Variable annual mining costs based on scheduled open pit production, LOM average of $2.22/t.*

Table 22-2 below summarizes the results of the economic analysis for the Project, both the Pre-Tax and Post-Tax results are shown. Figure 22-1 shows the estimated annual gold production by year that is used in the economic analysis.

Table 22-2 Summary of Economic Analysis

	Value	Units
Mill Feed	95.9	Mt
Au Grade	0.76	g/t
Au Produced	2,100	koz
Ag Grade	0.71	g/t
Ag Produced	876	koz
Waste Mined	382.9	Mt
Strip Ratio	4.0	t:t
Initial Capital	607	$M
Sustaining Capital	290	$M
Closure Capital*	130	$M
Cash Costs (LOM)**	696	US$/oz
AISC (LOM)***	801	US$/oz
Total Costs (LOM)****	1,068	US$/oz
Net Cash Flow (Pre-Tax)	1,471	$M
Pre-Tax (SMG)		
NPV, 5%	848	$M
IRR	25%	%
Payback	3.2	Years
Post-Tax (SMG)		
NPV, 5%	655	$M
IRR	22%	%
Payback	3.3	Years

** Closure capital costs are net salvage value costs of $30M.*
*** Cash Costs include all operating costs associated with the production and sale of gold, including royalties.*
**** All in Sustaining Costs (AISC) include Cash Costs as well as all sustaining capital costs related to the project.*
***** Total Costs include AISC as well as all initial capital and closure costs related to the project.*

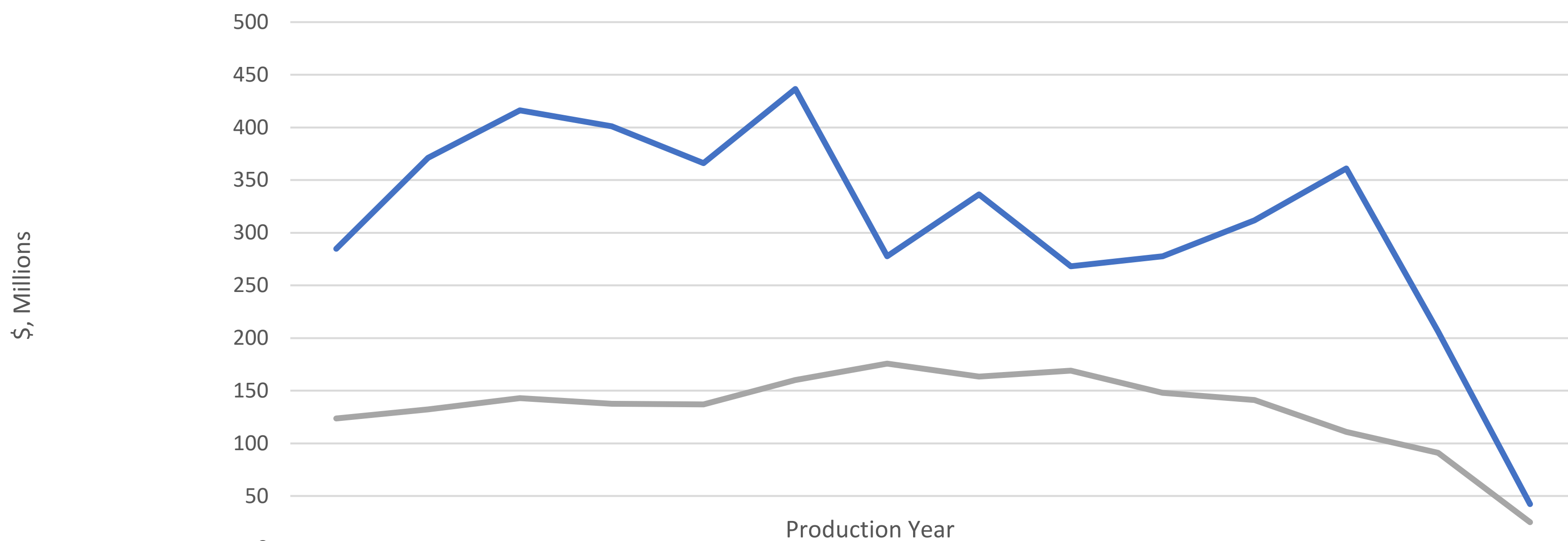


Source: Moose Mountain, 2021

Figure 22-1 LOM Gold and Silver Production

The following graph in Figure 22-2 shows by year:

- the estimated net gold and silver receipts
    - gross gold and silver receipts minus offsite charges: refining, transport, insurance, and royalty charges
- the estimated operating costs
    - mining, processing, TMF, water management and water treatment, and G&A costs



	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14
Net receipts (M$)	285	371	416	401	366	437	278	336	268	278	312	361	206	42
Operating Costs (M$)	124	132	143	138	137	160	176	163	169	148	141	111	91	25

Source: Moose Mountain, 2021

Figure 22-2 Net Receipts vs Operating Costs

Table 22-3 shows annual and LOM summaries of the cash flows in the economic analysis.

Table 22-3 Cash Flow Summary

	LOM	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15-20
Gold (koz)	**2,100**	0	0	137	179	201	193	177	211	134	162	129	134	150	174	99	20	0
Silver (koz)	**876**	0	0	64	67	72	64	68	73	65	66	61	65	65	66	60	18	0
Net Revenue (M$)	**$4,357**	$0	$0	$285	$371	$416	$401	$366	$437	$278	$336	$268	$278	$312	$361	$206	$42	$0
Operating Costs (M$)	**$1,859**	$0	$0	$124	$132	$143	$138	$137	$160	$176	$163	$169	$148	$141	$111	$91	$25	$0
Capital Costs (M$)	**$1,027**	$304	$303	$54	$27	$31	$27	$15	$35	$17	$17	$24	$29	$16	$12	$32	$36	$48
Pre-Tax Cash Flow (M$)	**$1,471**	**-$304**	**-$303**	**$107**	**$212**	**$243**	**$236**	**$214**	**$242**	**$85**	**$156**	**$75**	**$101**	**$155**	**$238**	**$83**	**-$19**	**-$48**
Taxes (M$)	**$313**	$0	$0	$2	$4	$5	$17	$40	$53	$9	$35	$14	$24	$43	$74	$18	-$11	-$17
Post Tax Cash Flow (M$)	**$1,159**	**-$304**	**-$303**	**$105**	**$207**	**$238**	**$219**	**$174**	**$189**	**$76**	**$120**	**$61**	**$77**	**$111**	**$164**	**$65**	**-$8**	**-$31**

The following graphs in Figure 22-3 through Figure 22-4 show, for each case, the economic result sensitivities to:

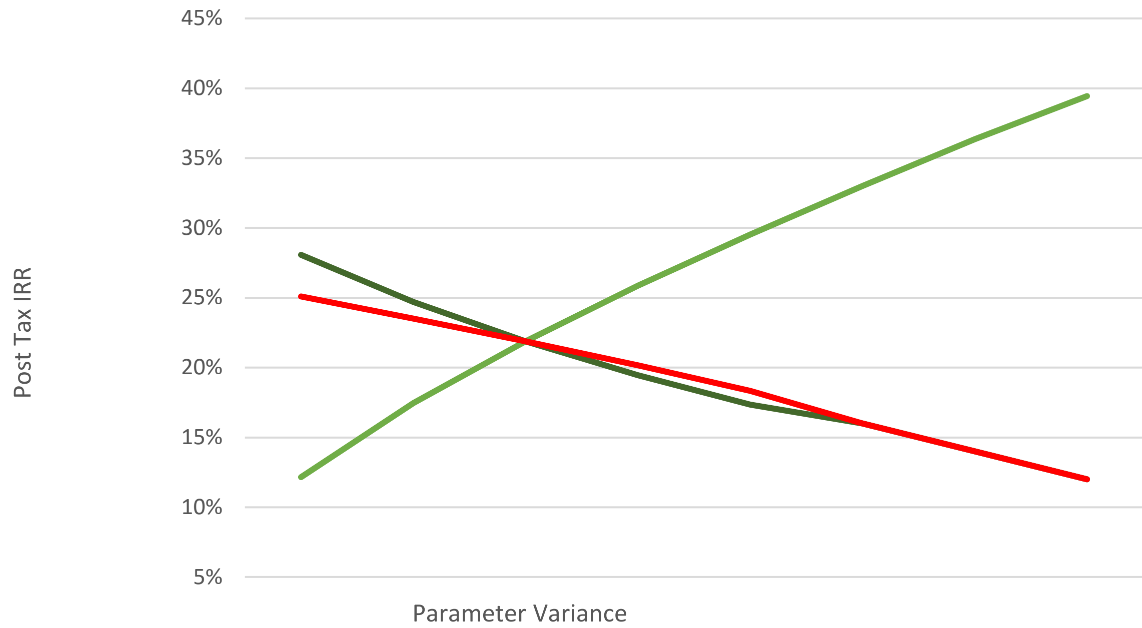
- Gold Price
- Project Capital Costs and
- Operating Costs (mining, processing, TMF, water management and water treatment, and G&A costs)

The Project is most sensitive to fluctuations in gold price assumptions, and less sensitive to variations in capital and operating costs. Foreign exchange rates and the gold grade is not presented in the sensitivity graph because the impact of changes in these inputs mirror the impact of changes in the gold price.

	-20%	-10%	0%	10%	20%	30%	40%	50%
Gold Price	252	465	655	841	1,026	1,211	1,396	1,580
Capital Costs	759	707	655	603	550	498	445	393
Operating Costs	812	733	655	575	495	412	322	230

Source: Moose Mountain, 2021

Figure 22-3 Sensitivity of Post Tax NPV (5% Discount Rate) to Project Inputs



	-20%	-10%	0%	10%	20%	30%	40%	50%
Gold Price	12%	17%	22%	26%	30%	33%	36%	39%
Capital Costs	28%	25%	22%	19%	17%	16%	14%	12%
Operating Costs	25%	24%	22%	20%	18%	16%	14%	12%

Source: Moose Mountain, 2021

Figure 22-4 Sensitivity of Post Tax IRR to Project Inputs

22.3 Post Tax Financial Analysis

A tax model was prepared by SMG to perform the post-tax economic evaluation of the Project with the inclusion of applicable income and mining taxes.

The components of the various taxes that will be payable on Spanish Mountain Profits over the 14-year mine life are shown in Table 22-4.

Table 22-4 Components of the Various Taxes

Tax Component	LOM Amount (M$)
Corporate Tax (Federal)	117.8
Corporate Tax (Provincial)	94.2
Less Investment Tax Credit (ITC)	(2.9)
Provincial Resource Tax	103.4
Total Taxes	**312.5**

The following general tax regime was recognized as applicable at the time of report writing.

22.3.1 Canadian Federal and BC Provincial Income Tax Regime

Federal and British Columbia provincial income taxes are calculated using the currently enacted corporate rates of 15% for federal and 12% for British Columbia provincial tax rate.

For both federal and provincial income tax purposes, capital expenditures are accumulated in Capital Cost Allowance (CCA) pools that can be deducted against mine income at different rates, depending on the type of capital expenditure.

Resource property acquisition costs and most other pre-production mine development expenditures are accumulated in the Canadian Development Expense (CDE) pool. The CDE is amortized against income at 30% on a declining balance basis.

Exploration expenditures other than those included in CDE are accumulated in the Canadian Exploration Expense (CEE) pool. The CEE is generally amortized at 100%, to the extent of taxable income from the mine.

Mining assets including processing machinery, equipment and facilities are accumulated in Class 41.2 and amortized at 25% on a declining balance basis once they are available for use.

Unused balances in CCA, CDE and CEE pools do not expire and may be carried forward to offset future taxable income. Non-capital losses generally can be carried forward for 20 years to offset future taxable income.

The tax model incorporates various tax pools, losses carry-forward and tax shields that can be reasonably expected to be available to offset future taxable income generated by the project. While tax rules allow such treatment for expenditures incurred by resource companies, the actual amounts of the available tax benefits may be different from what has been assumed. In addition, the tax model incorporates the tax impacts of certain expenditures accumulated by SMG (i.e., estimated balances in tax pools and unused corporate deductions) without including the actual expenditures in the project's cash-flow.

22.3.2 BC Mineral Tax Regime

The BC Mineral Tax regime is a two-tier tax regime, with a 2% net proceeds tax and a 13% net revenue tax.

The 2% tax is assessed on "net current proceeds", which is defined as gross revenue from the mine less mine operating expenditures including post-production development and reclamation costs. Hedging income and losses, royalties and financing costs are excluded from operating expenditures. The 2% tax is accumulated in a Cumulative Tax Credit Account (CTCA) and is fully creditable against the 13% tax.

All capital expenditures, both mine development costs and fixed asset purchases, are accumulated in the Cumulative Expenditures Account (CEA), which is amortized at 100% against the 13% tax.

The 13% tax is assessed on "net revenue", which is defined as gross revenue from the mine, less mine operating expenditures, less any accumulated CEA balance. As such, the 13% tax is not assessed until all pre-production capital expenditures have been amortized.

Notional interest of 125% of the anticipated federal bank rate, based on long term average, is calculated annually on any unused CEA and CTCA balances and is added to these pools.

The BC Mineral Tax is deductible for federal and provincial income tax purposes.

23 ADJACENT PROPERTIES

There are no active exploration properties immediately adjacent to the Spanish Mountain Property. The Property is in an area that has seen active past exploration and mining activity for alkaline porphyry copper-gold deposits that are completely segregated from the Spanish Mountain Gold Property and are not in any way indicative of the mineralization on the Property.

Currently, the most advanced property in the area is Imperial Metals' Mount Polley Mine, which is alkalic porphyry copper-gold deposit located about 15 km to the west. As of December 31, 2018, the deposit had measured and indicated resources of 194Mtonnes grading 0.29% copper and 0.29 g/t gold (Imperial Metals website).

The QR Mine is a propylitic gold skarn located 24 km northwest of the Property. As of July 2009, the West Zone had a measured resource of 40,000 tonnes grading 3.65 g/t Au and an Indicated resource of 479,000 tonnes grading 4.18 g/t Au, all at a cut-off grade of 2.0 g/t Au (Fier et al., 2009).

Various placer properties and operations on placer leases exist in and around the Likely area. Very little public information is available about the gold content in the placer deposits.

The QP, William Gilmour, P.Geo., has been unable to verify the information stated above and the information stated above is not necessarily indicative of the mineralization on the Spanish Mountain Gold Property.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution Plan

24.1.1 Project Milestones

Table 24-1 presents the milestones for the Project. Seasonal restrictions will ultimately dictate the construction schedule for construction, so it is assumed the start to be Q1, 2025. The target for completion of all construction and pre-commissioning works is end of Q3 2026.

Table 24-1 SMG Milestones

Milestones (Completion Dates)	Milestone by Calendar Date	Milestone Completion Date
BC Hydro System Impact Study (SIS)		
BCH Approval of SMG Powerline Design/Drawings	1-Dec-24	Q4 Y-3
Environmental Permits		
Approval of key construction permits	08-Jan-25	Q1 Y-2
Mining		
Commence Open Pit Pre-development and Mining	01-Mar-25	Q1 Y-2
Engineering and Procurement		
EP Contract Award	24/12/24	Q4 Y-3
Complete Process Design and Drafting	14-May-26	Q2 Y-1
Equipment Procurement (General Equipment)	13-Mar-25	Q1 Y-2
Concrete, Structural Steel and Architectural Design	16-Apr-26	Q2 Y-2
Mechanical Piping Design	18-Mar-26	Q1 Y-2
Process Plant Electrical & Instrumentation Design	16-Apr-26	Q2 Y-2
Construction – Process Plant and Infrastructure		
Site Establishment	12-April-25	Q2 Y-2
Complete Concrete Works	27-Oct-25	Q4 Y-2
Complete SMP Installation	18-Sep-26	Q3 Y-2
Complete Electrical and Instrumentation	18-Sep-26	Q3 Y-2
Construct Main Substation and 138 kV powerline	27-Sep-25	Q3 Y-2
Construct Surface Infrastructure	9-Nov-26	Q4 Y-2
Construction – Tailing Storage Facility		
WT Passive Treatment Engineering	03-Jan-25	Q1 Y-2
TSF Engineering	03-Jan-25	Q1 Y-2

Milestones (Completion Dates)	Milestone by Calendar Date	Milestone Completion Date
Water Management Engineering	03-Jan-25	Q1 Y-2
Install Tailings Distribution System	14-Aug-26	Q3 Y-1
Install Reclaim and Surplus Water Systems	14-Aug-26	Q3 Y-1
Construction – Water Treatment Facilities		
Construct Water Treatment facilities	14-May-25	Q2 Y-2
Construction – Access Roads		
Construct Access road from Mill to Spanish Lake Road	1-Mar-25	Q1 Y-2
Construct Access road north of mine facility re-route Spanish Lake Road	1-Mar-25	Q1 Y-2
Commissioning – Process Plant		
General Commissioning	21-Oct-26	Q4 Y-1
First Gold-Silver Doré	28-Jan-27	Q1 Y1
Complete Closure Activities	30-Jun-46	Q2 Y20
Project Completion Date	30-Jun-46	Q2 Y20

24.1.2 Scope and Project Approach

The scope of the study includes all the work required for the SMG mine.

The strategy is to construct mill facilities, ancillary facilities, and design/construct the tailings storage facility (TSF) and Water Management infrastructure to accommodate the mill throughput tailings from the SMG process plant and reclaim water system. Design and construction of a new water treatment plant (WTP) and associated passive treatment of water bodies post closure.

BC Hydro will establish a new 138 kV Substation near Highway 97, to be known as SMMX Substation. The new substation will be fed at 230 kV from existing BC Hydro line 2L95. The new substation will be owned and operated by BC Hydro. Everything downstream of the Point of Intersection (POI) will be owned and operated by Spanish Mountain.

A new 138 kV transmission line will be installed between the POI and the receiving substation at the Spanish Mountain mine site. The transmission line will be single pole type. The transmission line will generally follow the road, to a point just west of the town of Likely, then it will be routed north, around Likely, to the mine site. The transmission line will be owned and operated by Spanish Mountain.

A new 138kV Receiving Substation including 138kV disconnect switch, circuit breaker, transformer and capacitor bank will be constructed. On-Site distribution will be installed at 13.8 kV

The project schedule was developed with input from the mining team, the processing and surface infrastructure teams to synchronize the production ore and interactions with the surface infrastructure. This also involved input on the TSF, and WTP installations.

The process plant and infrastructure schedule was developed in conjunction with a regional contractor to assess the installation of process equipment.

The execution plan at this stage should be considered developmental and will undergo several optimization phases prior to the approved execution of the Project.

The main aspects of the Project include the following:

- Open Pit mine development work.
- Pipelines will be installed from the process facilities to the TSF. A Rougher tailing and separate Cleaner tailing pipeline will be installed from the mill building to separate compartments in the TSF. The supernatant water from the Rougher and Cleaner compartments of the TSF will be pumped to the WMP. Surface run-off and other surface water catchments will be directed into the WMP. Water stored in the WMP will be pumped to the WTP, and treated water from the WTP will be discharged into Cedar Creek.
- Construction of the major water diversions, channels/ponds on site.
- Design a two compartment TSF to separate the resulting two tailings streams from the SMG process facilities.
- Construction of on-site access/haul roads from the Open Pit to the primary crushing facility and adjacent Waste Rock Dumps.
- Construction and commissioning mill facilities.
- Construction of non-process ancillary facilities and site services
- Construction a new access road north of the mine facility (Spanish Lake road).
- Construction and commissioning of passive and active water treatment facilities.
- Construction of a new 138 kV transmission power supply from a new Substation at Highway 97 to the SMG site.
- Construction of a receiving Substation on-site located close to the mill building.
- Installation of powerlines to service the mining activities, process facilities, non-process facilities, TSF facilities (including the Tailing Reclaim System), and the water treatment facilities.

24.1.3 General Principles

The following are a list of general principles to consider for the SMG project:

- Safety will be a top priority in the execution of the SMG. SMG will ensure that safety, health, and environmental management is at the forefront of the Project execution.

- Early works will consist of construction of a temporary construction camp, 138kV off-site transmission line, receiving substation, access road construction and bulk earthworks for the plant site.
- Establish permanent infrastructure early, to the extent practical, to minimize costs of temporary construction facilities.
- Negotiate contracts with Suppliers, Contractors, and Engineering Procurement and Construction Management (EPCM) services with a proven track records in mine developments and similar construction projects.

24.1.4 Access—Construction Materials

Required construction materials are planned to be transported to site via existing all-weather roads.

24.1.5 Personnel

SMG assumes that local labour is readily available from the surrounding area. However, SMG has made provision for a temporary construction camp close to the existing mill facility. Some construction personnel will be accommodated in the temporary construction camp and some in the existing exploration camp. Access for the personnel will be surrounding local and national roads.

The temporary construction camp will not be dismantled after the construction phase. Depending on demand, some rooms may be upgraded as required to accommodate additional staff.

The existing exploration camp will also be utilized as a permanent camp facility for the mine life.

24.1.6 Construction Strategy

Where possible, the contracting strategy will specify several smaller work packages to allow local contractors participation in the Project.

As a special note for major construction fills, crushed waste rock will be free issued to the construction areas from the Crushing Plant at the TSF area. Costing allowances have only been included for hauling, placement, and compaction costs.

The general approach to construction execution strategy includes the following key elements:

- Extensive front-end construction planning to identify priorities and increase efficiency in engineering, procurement, and construction.
- Construction will be planned and executed by sub-area and trade according to construction work package philosophy.
- Constructability analysis and input in the design phase to influence engineering, minimizing construction risks and optimizing project costs and schedule.
- Major construction equipment such as crushing plants and large cranes shall be supplied by Contractors.
- It is assumed that most of the construction labour force will come from the local region.

- The construction labour is expected to work 10 hours per day and 7 days per week for the duration of the construction and commissioning phase, while abiding within the limits of British Columbia labour law.

The schedule shown in Figure 24-1 outlines the mining production sequence for the project:

- Costs of the Project are heavily dependent on performing work in the appropriate season.
- All earthworks are best executed in spring/summer seasons.
- Concrete work is best done in the spring/summer seasons to avoid winter heating and hoarding costs and improve outside work productivity. Concrete will be supplied by a local concrete supplier from Likely.
- Pre-assemble as much equipment as possible, e.g., air compressor skids or modules.

24.1.7 Planning and Scheduling

The current project execution schedule was developed for the SMG project, as shown in Figure 24-1. The schedule was developed with interaction between the mining, processing, and surface infrastructure team to check for timing of the production ore and interactions with the surface infrastructure and tailings facilities scopes of work.

The process equipment schedule considered the early ordering of equipment using the designs developed for the feasibility study in which key long lead items can be tendered. These long lead items will be reviewed to the point of a technical recommendation for SMG.

The critical path of the Project execution which includes the following stages:

The schedule presented in Figure 24-1 outlines the SMG timeline. Construction activities are dependent on receiving approved permit applications.

Task Name	Start	Finish
SMG Execution Schedule	**Tue 06/10/20**	**Tue 23/02/27**
Key Milestones	**Tue 06/10/20**	**Sat 05/12/26**
Contract Award	Tue 06/10/20	Tue 06/10/20
Key Construction Permits	Wed 08/01/25	Wed 08/01/25
First Dore	Sat 05/12/26	Sat 05/12/26
Prefeasibility Study	**Tue 06/10/20**	**Mon 14/03/22**
Feasibility Study	**Mon 14/03/22**	**Fri 06/10/23**
EPCM	**Tue 24/12/24**	**Tue 23/02/27**
Detailed Engineering	**Tue 24/12/24**	**Thu 14/05/26**
Engineering	**Thu 02/01/25**	**Thu 14/05/26**
Procurement & Contracts	**Thu 13/03/25**	**Fri 26/06/26**
H.O. Site Support	**Mon 23/06/25**	**Thu 24/09/26**
Site Mobilization	**Sun 09/03/25**	**Sat 12/04/25**
Site Establishment	**Sat 12/04/25**	**Fri 10/10/25**
Construction Plant Site	**Mon 11/08/25**	**Fri 18/09/26**
Commissioning	**Fri 18/09/26**	**Sat 05/12/26**
Ramp-up	**Sat 05/12/26**	**Tue 23/02/27**
Off-Site Electrical	**Fri 03/01/25**	**Mon 27/10/25**
On-Site Electrical	**Fri 31/01/25**	**Thu 05/02/26**
Water Treatment Facilities	**Fri 03/01/25**	**Wed 23/07/25**
TSF facility and Water Mgt	**Fri 03/01/25**	**Tue 08/12/26**

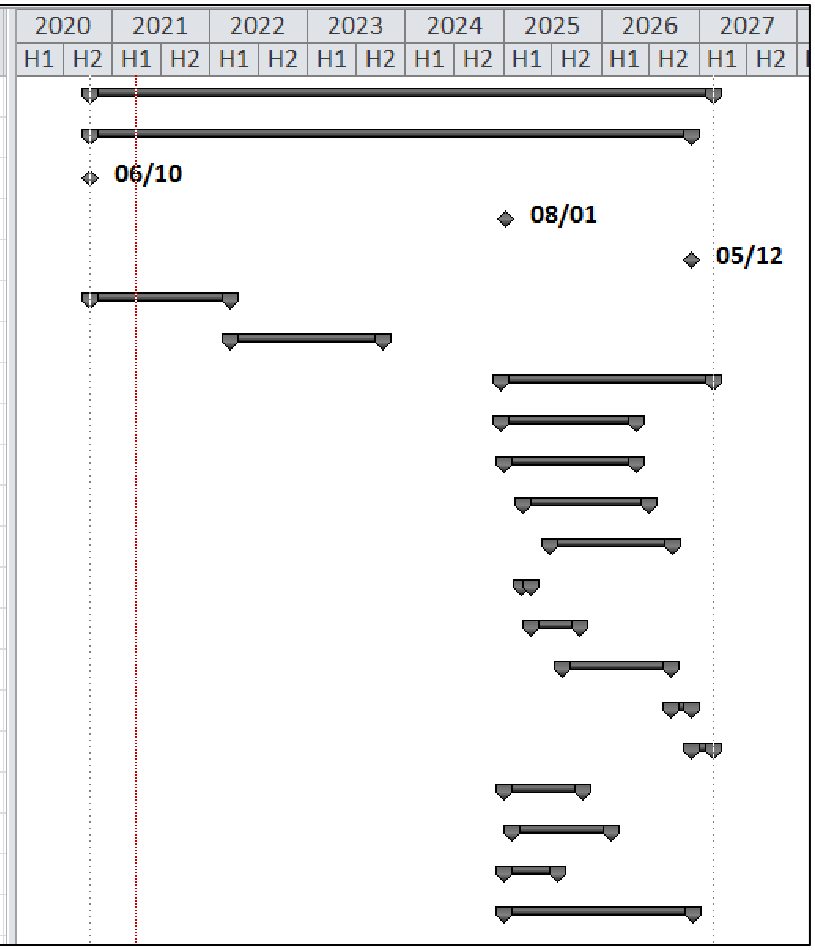


Figure 24-1 SMG Overall Project Schedule

The current plan will be to complete interim studies and fieldwork before executing a Feasibility Study by Q4 2023 and to proceed to detailed engineering. This will allow SMG to be able to start early works at site in Q1 2025 (Y -2). This work will include early construction of the earthworks for the water treatment facilities and the installation of 138 kV transmission line from Highway 97 to the new Main Substation.

Upon Project approval starting in Q1 of Y-2, the balance of the facilities will be constructed. The total Project is expected to require 24 months, resulting in Project start-up in Q1 of Year 1 (2027).

25 INTERPRETATION AND CONCLUSIONS

This Prefeasibility Study represents an economically viable, technically credible, and environmentally sound mine development plan for the Project. This Prefeasibility Study level fourteen-year life-of-mine (LOM) plan has positive economics. Industry-standard mining and processing methods were used in the study, and the Qualified Persons (QPs) are not aware of any fatal flaws that encumber the SMG from undergoing further economic studies, permitting, financing, and ultimately development. The QPs recommend that the Project proceed to permitting, additional fieldwork, trade-off studies, and a feasibility study.

The most significant potential risks associated with the Project are uncontrolled mining dilution, operating and capital cost escalation, permit acquisition, reduced metallurgical recoveries, unforeseen schedule delays, changes in regulatory requirements, the ability to raise financing, exchange rate, and metal prices. These risks are common to most mining projects, many of which may be mitigated, at least to some degree, with adequate engineering, planning, and proactive management.

Further conclusions and interpretations regarding the next phase of the SMG are given below.

25.1 Geology, Mineral Resources, and Mineral Reserves

25.1.1 Mineral Resource Estimate

The estimation of the Spanish Mountain Gold Project's Mineral Resources for the Prefeasibility Study was carried out with the same drillhole data and geologic model as for the 2019 PEA study, as no new assays were available and no new geologic information was acquired since then. All other aspects of the Mineral Resource estimation process were re-visited for the Prefeasibility Study. An ordinary kriging grade interpolation technique with capped composites was selected as the grade estimation approach for gold and silver. A multiple indicator kriging method for gold and inverse distance cubed method for silver were previously used for the 2019 PEA study.

Approximately 20% of the drillholes utilized for this study are from reverse circulation drilling (RC), with the remainder of holes being diamond drillholes (DDH). Comparative statistics between the RC and DDH gold assays have shown higher average gold grades from the RC holes. Further investigation did not indicate downhole contamination from the RC drilling. Marc Jutras is satisfied that the much larger sample derived from the RC drilling in an environment of coarse gold in friable material provides a better grade approximation.

The higher gold grade variability denoted by the high coefficients of variation from the 1.5m composites could be indicative of a grade estimation technique where higher grades are treated differently than lower grades, such as a multiple indicator kriging method. However, from the capping analysis the high coefficients of variation were reduced to acceptable levels for an ordinary kriging method to be utilized.

An independent validation of the drillhole database was carried out by Marc Jutras prior to the estimation of the Mineral Resources. In this exercise approximately 10% of the gold assays were checked against the assay certificates. Similarly, approximately 10% of the drillhole collar coordinates and downhole surveys were checked against the drill logs. Overall, no significant errors were found and the drillhole database was deemed valid for the estimation of Mineral Resources.

Marc Jutras is satisfied that the current geology model is adequate for the estimation of Mineral Resources.

The abundance of drillhole data at a relatively close spacing allowed for conclusive variogram models for gold and silver. This brings greater certainty to the assessment of the gold and silver grade continuity.

The satisfactory results obtained from the various validation tests for the gold and silver estimates indicate an adequate representation of the Spanish Mountains Gold Project's Mineral Resource, based on the current geologic understanding and available information.

25.2 Mineral Processing and Metallurgical Testing

25.2.1 Open Pit Deposit

The Spanish Mountain Gold completed numerous comminution and metallurgical testwork programs to support the PFS. This included head grade analyses, a full suite of comminution tests, flotation, gravity separation, and leach tests and cyanide detoxification.

Tests were performed on mineralization that is representative of the 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 over the life of mine. The grade variability samples gold and silver grades ranged from 0.15–3.25 g/t Au and 0.5–4.4 g/t Ag.

25.2.2 Primary grind

Selection of primary grind size should be revisited in future studies as the P80 size of 180 µm was selected at a time when the lower gold price did not justify finer grinding.

Bulk mineralogy on select composites showed that pyrite was the main sulfide mineral present, representing between 0.5-2.5% by mass. Sphalerite and chalcopyrite were also present in relative order of abundance.

Comminution testing showed that the materials tested are highly variable in competency. The breakage data showed the ore can be classified as competent and moderately hard, and moderately abrasive with SAG Mill Comminution (SMC) tests ranging from 26-51.7. Conventional Bond tests showed significant variation in hardness, with Bond rod mill work indices ranging 12.4-17.5 kWh/t and Bond ball mill work indices ranging from 10.9-16.7 kWh/t.

Rougher flotation tests showed high sulfide recovery was generally achieved within 8 minutes of flotation time. Flotation recoveries to cleaner concentrate ranged 80-92% for gold, 25-55% for silver.

High leach recoveries were achieved when leach feed was reduced to <0.5% TOC and after regrinding.

Overall plant recoveries for gold are predicted to range from 85–92% for head grades ranging from 0.6-1 g/t Au. Overall plant recoveries for silver are predicted to range from 38–42%.

Cyanide detoxification tests reduced WAD cyanide levels to 1.5 mg/L with moderate reagent consumption rates.

The primary gravity concentration circuit was reinstated as part of the process flowsheet optimisation.

25.3 Mining Methods and Reserves

Mineral Reserves were determined and calculated using appropriate mining methods and cost estimates to industry standards

25.3.1 Mineral Reserves

Proven and Probable Mineral Reserves have been modified from Measured and Indicated Mineral Resources at Spanish Mountain Gold. Inferred Mineral Resources have been set to waste. The Mineral Reserves are supported by the 2021 Prefeasibility Study.

Factors that may affect the Mineral Reserve estimates include metal prices, changes in interpretations of mineralisation geometry and continuity of mineralisation zones, geotechnical and hydrogeological assumptions, ability of the mining operation to meet the annual production rate, operating cost assumptions, process plant and mining recoveries, the ability to meet and maintain permitting and environmental licence conditions, and the ability to maintain the social licence to operate.

25.3.2 Mining

Reasonable open pit mine plans, mine production schedules, and mine capital and operating costs have been developed for the Mineral Reserves estimates at Spanish Mountain Gold.

Pit layouts and mine operations are typical of other mountainous open pit gold operations in Canada, and the unit operations within the developed mine operating plan are proven to be effective for these other operations.

The mine plan supports the cash flow model and financials developed for the Prefeasibility Study.

25.4 Infrastructure

25.4.1 Tailings and Water Management

Conclusions associated with tailings and water management for the SMG project include:

- The TSF has the capacity to store the tailings and PAG waste rock produced for the 14-year mine life.
- There is sufficient material available to construct the TSF embankment with most of the construction material being sourced from the pit stripping activities.
- The site investigation work completed at the TSF is sufficient for a PFS study.
- The two tailing streams produced in the mill are effectively managed at the TSF. The rougher tailings will be discharged from spigots located at the embankments and the west side of the TSF forming extensive drained tailings beaches. The cleaner tailings and detox tailings stream will be pumped and discharged sub-aqueously into the PAG Cell, a separate location within the facility, referred to as the PAG Cell.

- Managing the PAG tailings and waste rock in a separate internal cell facilitates the development of large tailings beaches in the TSF and the subaqueous management of the PAG materials in the PAG Cell.
- Site runoff is stored in the Water Management Pond which has been sized to store the site runoff from the 95th percentile wet year assuming a water treatment rate corresponding to approximately the 80th percentile of annual site runoff.
- The site plan includes diversions where required to minimize the amount of non-contact water entering the project site.
- The TSF design allows for effective reclamation of the tailings surface with minimal surface ponding post closure.

25.4.2 Water Treatment Process

Water treatment will be required during all years of the Project, except for the first Pre-Production Year (Y-2) when best practices and water management will be employed to address suspended solids generated by construction. Beginning inY-1, active treatment will be employed to remove deleterious parameters from site waters. Active treatment will continue through Y18, transitioning to passive treatment during the Active Closure period. The treatment systems are conservatively designed to meet applicable water quality standards at the end-of-pipe; no assimilative capacity in the receiving streams (Cedar and Spanish Creeks) is assumed.

Spanish Mountain Gold will construct a WTP to actively treat influent from the Site. The WTP will incorporate a variety of process technologies including oxidation, settling and clarification, microfiltration (MF), denitrification, and RO. The WTP effluent will discharge to Cedar Creek. Design and construction of a 15,000 m^3/day WTP will occur in Y-2, with operations beginning in Y-1. The WTP will initially be equipped to treat nitrogen compounds (blasting residuals) in Y-1. In Y1, the WTP will be fitted with additional treatment equipment to address MIW generated during Production including removal of parameters such as sulphate and metals. The WTP capacity will be expanded to 18,000 m^3/day in Y6 as the mine footprint expands. The WTP will continue to operate into the Active Closure period. The WTP will be decommissioned and removed from the site in late Y18.

Separate PTSs will be constructed to individually address seepage from the North RSF, the West RSF, and the North SCP. The PTSs will include combinations of iron terraces, BCRs, and aerobic polishing wetlands. Effluent from the West RSF and North SCP PTSs will be directed to Cedar Creek, while that from the North RSF will be directed to Spanish Creek. Design of the North RSF, West RSF, and North SCP PTSs will occur in Y22 and Y23, with construction occurring in Y24. These three PTS will be operated at 25% and 75% capacity in Years Y25 and Y26, respectively, becoming fully operational beginning Y27. Primary passive treatment of the Pit Lake will occur in-situ, with a downstream iron terrace treatment unit constructed for polishing before discharging to Spanish Creek. Design of the Pit Lake PTS will occur in Y29 and Y30, with construction occurring in P13 as active mining ceases. The timing of the Pit Lake PTS construction is considered conservative because the Pit Lake is predicted to not fill its spill point into Spanish Creek until Y25.

25.4.3 Off-Site Infrastructure

No closure costs were applied to the 138kV electrical transmission line and the on-site receiving substation as they could be of benefit to the local community.

25.5 Geochemistry

Geochemical characterization for the PFS was based on the 2012 SRK study that indicated:

- The proportion of rock with ARD potential at Spanish Mountain is relatively low.
- The potential for metal leaching at neutral pH however was identified regardless of ARD potential.
- The rougher tailings generated from metallurgical testing had low ARD and metal leaching potential, and the cyanide leach tailings indicated a high potential for ARD and metal leaching.
- Results indicate that management measures will be required to reduce the generation and effects of ARD/ML.

25.6 Environmental Studies, Permitting, and Social or Community Impact

The environmental and community aspects of the Project have been well studied and are understood. Ongoing and updated studies continue to add clarity to the baseline condition and will be used as a foundation to re-enter the environmental assessment and permitting processes. There are no known environmental or social constraints that could materially impact the issuer's ability to extract the mineral resources or mineral reserves.

25.7 Capital and Operating Cost Estimates

The initial capital costs can be optimized by using one Engineering and Procurement (EP) consultant incorporating the process and infrastructure designs. Construction costs can be optimized by using an experienced local Construction Management (CM) contractor to construct the process building and equipment and required infrastructure.

25.8 Economic Analysis

Initial capital cost for construction of the Project is estimated to be $607.2M. Sustaining capital requirements over the 14-year mine life are estimated to be $290.5M. Closure costs, net of salvage values, are estimated to be $129.6M.

Operating costs for the Project are estimated at a unit cost $19.38/t milled.

Closure Costs of $159.6M and a Salvage Value of $30M.

The post-tax NPV5% is $655M, and post-tax IRR is 22%. The projected payback is 3.3 years.

The all-in sustaining cost is US$801/oz. Au. The LOM production is estimated at 2.1M oz gold and 0.9M oz silver.

The project is most sensitive to fluctuations in gold price and foreign exchange rate assumptions, and less sensitive to variations in capital and operating costs.

26 RECOMMENDATIONS

The Project is well suited for a potential mining operation. The Prefeasibility Study-level 14-year life-of-mine (LOM) plan has positive economics, and it is recommended that the Project proceeds to permitting and feasibility study. Further recommendations regarding the next phase of the SMG are discussed below.

26.1 Geology, Mineral Resources, and Mineral Reserves

26.1.1 Mineral Resource Estimate

During the drillhole database validation exercise, a survey of drillhole collars carried out in 2008 by Allnorth indicated different collar coordinates for 48 holes. From further investigation it was found that these holes were re-surveyed after this initial survey and that the collar coordinates in the drillhole database are the correct ones. Survey checks for a few drillholes carried in December 2020 have confirmed the current coordinates. It is recommended that all the 48-hole collars be re-surveyed during the 2021 summer months and documented to further confirm their collar coordinates.

Marc Jutras recommended that the broader geologic units could be refined for future updates.

A drilling campaign was carried out in November and December of 2020 and was not incorporated into this Mineral Resource estimate due to delays in the assaying process. It is recommended that an update of the Mineral Resource estimate be carried out with these additional drillholes.

The budget for the above recommendations by Marc Jutras is approximately $62,000, and is detailed in the following Table 26-1:

Table 26-1 Recommendation Work Details

Task	Description	Cost
Re-Survey of Drill Holes	Confirmation survey of 48 drill hole collars from the Allnorth surveying campaign of July 2008	$4,500
Update of the Mineral Resource Estimate with the Nov/Dec 2020 Drill Holes	Update of the geology model with new drill holes and refining the interpretation of the geologic units.	$19,500
	EDA update: compositing, capping, statistical plots	$4,000
	Redo variographic analysis	$8,000
	Au and Ag grade interpolation	$6,000
	Validation of Au and Ag grade estimates	$6,000
	Tabulation of mineral resources	$4,000
	Writing of report	$10,000
	Total	**$62,000**

26.2 Mining

MMTS recommends that following testwork and analysis to advance the project to Feasibility level engineering designs:

- The following geotechnical and hydrogeological testwork and analysis to bring the pit slope designs to a Feasibility level design:
    - Geotechnical logging of core holes targeting shallow faults, which are currently limiting the PFS design.
    - Large-scale geotechnical outcrop mapping to characterize the large-scale roughness, spacing and persistence of the controlling fault sets.
    - Delineate watershed areas directing surface water into the proposed pit and investigate the drainage features evident near the east and west walls.
    - Investigation and numerical simulation of the degree of hydraulic connection between the north end of the open pit and Spanish Creek. Conduct ground geophysics at the base of the Spanish Creek valley, to better characterize the depths of alluvial sediments and fractured bedrock.
    - Investigation and numerical simulation of the depressurization requirements of the updated open pit.
    - A detailed geological model, including folding, faulting, and a reconciliation of the various interpretations on the deposition and tectonic history of the deposit.
- Geotechnical analysis of the foundations identified for the WRSF's.
- Condemnation drilling of the footprints identified for the WRSF's, and site infrastructure should be carried out.
- Execute a grade control drilling and interpretation program, or possibly even a bulk sample program, in selected areas of the deposit that are planned to be mined for initial mill feed. The resultant tonnes and grade from this interpretation should be compared to the equivalent area resource modelled tonnes and grade. Results to be incorporated in ongoing grade control strategy and mine planning.
- Drill and blast testing to be carried out by drilling vendors and local explosives suppliers by analysing local rock types and conditions to assess the achievable drill penetration rates, optimal explosives mix and target powder factor for use in this operation.
- Updating of all mine planning work to incorporate results from other recommended studies, including optimization studies for pit limits and mine scheduling, and various operational trade-off studies (contractor vs. owner fleet, lease vs. purchase, etc.).

The approximate costs of the above recommendations are as follows:

- Geotechnical drilling and hydrogeological pumping tests = $2.8M
- Geotechnical and hydrogeological field work, lab work and engineering for FS level design = $1.4M
- Condemnation drilling = $1.0M
- Grade control drilling or bulk sampling = $3.0M
- Remaining engineering, including Feasibility study and analysis = $0.3M

26.3 Processing Facilities and Metallurgical Testwork

The Metallurgical testwork recommended in support of feasibility study is as follows:

1. Primary grind size assessment
2. DFR rougher pilot study on gravity tails for confirmation of predicted performance (recovery, mass pull and TOC rejection, and mineral analysis by size of rougher concentrate)
3. Bench scale cleaner tests on DFR rougher concentrate to evaluate:
   - second cleaner stage included or excluded
   - open vs closed circuit for cleaners
   - gold by size analysis of cleaner tailings
   - gravity scavenger test performance
4. Bulk concentrate generation to support equipment sizing and reagent consumption, specifically:
   - Determination of vendor specific energy for regrind
   - Dynamic settling tests – cleaner tailings and concentrate after regrinding
   - Slurry rheology - concentrate slurry at P80 ± 20 microns and 35-50% solids
   - Leach time evaluation
   - Oxygen uptake test
5. Variability sampling to provide spatial coverage of the ore body, testing at optimised flowsheet conditions to include the following:
   - Chemical characterisation
   - Determination of comminution properties (SMC and BBWI, RWi, Ai data)
   - Gravity/Flotation/scavenger gravity/regrind/CIL performance
   - Grade vs recovery relationships for varying S, TOC, and Au grades
6. Tailings characterisation – flotation and leach tailings (determination of chemical and mineral composition of solids, chemical composition and water quality parameters for solutions, and cyanide speciation of detox solution)

Five engineering trade-off studies are recommended as follows:

- Primary grind size evaluation
- Regrind size evaluation and technology selection – to inform test work program
- Primary gravity circuit evaluation – to inform test work program
- Scavenger gravity circuit evaluation – to inform teste work program
- Trade-off of flotation recovery vs mass pull and corresponding concentrate processing circuit capital and operating costs

The cost of testwork (excluding drilling) is expected to range between $ 400,000 to $450,000 dependant on the final variability sample selection. Testwork management costs are estimated at $75,000, and engineering trade-off studies at $50,000.

The cost for a feasibility study covering the process and infrastructure scope is approximately $1.8M.

26.3.1 On-Site Infrastructure

The QP recommends a site-wide civil geotechnical investigation to verify ground bearing pressures, soil parameters, ground dynamic modulus of elasticity, and settlement parameters: the approximate cost of this work is $250,000.

A route survey report is recommended for the next phase, prior to construction, at an approximate cost of $30,000.

26.3.2 Off-Site Infrastructure

- Confirm viability of installing 138 kV pole line alongside of road between BC Hydro substation and site. (Budget estimate $20,000)
- Investigate possibility of installing 138 kV pole line through Likely (thus saving 10 km of transmission line). (Budget estimate $20,000)
- Issue detailed line design to electrical contractor.
- Perform a LIDAR survey of the 138kV transmission line route. (Budget estimate $20,000)
- BCH ongoing engineering co-ordination related to the System Impact Study (Budget estimate $5000)
- Off-Site electrical Feasibility Study estimated budget $70,000
- Investigate local community benefits of the continuous service of the 138kV overhead distribution line and receiving site substation. Estimated budget for the investigation $15,000

26.3.3 Tailings Storage Facility and Surface Water Management

Les Galbraith recommends the following for the advancement of the SMG tailings and water management:

- Complete additional site investigation programs and laboratory testing to support the level of detail required for future studies (Estimated budget - $3.0M).
- Completing a site-specific Seismic Hazard Analysis (SHA) to confirm the seismic design parameters (Estimated budget - $0.1M).
- Develop geochemical source terms and a site water quality model, (Estimated budget - $0.10M).
- Completing a surficial landform mapping desktop study. This requires higher resolution LiDAR data that what is currently available for the site (Estimated budget - $0.10M).

26.3.4 Water Treatment Process

The water treatment processes planned for the Project are conventional, proven technologies that are commonly implemented to treat mine and tailings water produced at open pit gold mine operations. The projected volumes of water to be treated, both by active and PTSs, are very large. Additional studies and design advancements to reduce the generation of MIW could decrease the volume of impacted water to be treated, which would lower both CAPEX and OPEX. Examples include the use of dewatering wells in advance of pit development, construction of less permeable caps on rock storage facilities and/or use of impermeable interlayers to decrease infiltration, and progressive reclamation and optimized water management to decrease the volume of surface water requiring treatment. In addition, source control and/or selective waste rock handling to reduce the concentrations of parameters requiring treatment could lower both CAPEX and OPEX. An example includes the use of nature-friendly bactericides on rock storage facilities and/or select pit wall areas to inhibit the microbial oxidation of pyrite, which could potentially reduce the need for expensive sulphate treatment. Advances to the Project's Water Balance and Water Quality Models would also provide improved information on the timing and magnitude of parameters requiring treatment; this information can be used to optimize the water treatment strategies and improve CAPEX and OPEX estimates. Finally, further assessment of the demolition, haulage, disposal/salvage of the WTP should be performed to refine the closure costs associated with the structure and associated water treatment equipment.

The estimated costs of studies for the next phase to Feasibility Study level are as follows

- the active WTP $275,000
- the passive treatment system designs $75,000, and
- the sulphate reduction studies is $190,000.

26.4 Geochemistry

Recommended geochemistry related tasks for post-PFS studies to support feasibility and permitting for the Spanish Mountain Gold Project are as follows:

- Expanded static and humidity cell test characterization program on waste rock from the eight geological domains within the proposed open pit (Currently in progress).
- Addition of a saturated column to the waste rock characterization program to assess the loading from submerged potentially acid generating (PAG) waste rock.
- Geochemical characterization of ore feed, tailings and process water generated from the upcoming pilot plant metallurgical testing program.
- Static test characterization of overburden and borrow sources that may be used in construction/infrastructure for the project.
- Development of source terms that will be used as inputs into the site-wide water and load balance (undertaken by others) to generate water quality predictions to support water treatment evaluations and permitting.

The estimated budget for these recommendations is $370,000.

The Recommended geochemistry related tasks for post-PFS studies to support feasibility and permitting for the Spanish Mountain Gold Project are as follows:

- Expanded static and humidity cell test characterization program on waste rock from the eight geological domains within the proposed open pit (Currently in progress).
- Addition of a saturated column to the waste rock characterization program to assess the loading from submerged potentially acid generating (PAG) waste rock.
- Geochemical characterization of ore feed, tailings and process water generated from the upcoming pilot plant metallurgical testing program.

- Static test characterization of overburden and borrow sources that may be used in construction/infrastructure for the project.
- Development of source terms that will be used as inputs into the site-wide water and load balance (undertaken by others) to generate water quality predictions to support water treatment evaluations and permitting.

The estimated budget for these recommendations is $370,000.

26.5 Environmental Studies, Permitting, and Social or Community Impact

KP recommends that environmental and social studies continue and that the Project re-enter the environmental assessment process in 2021. Estimated budget for these activities is $3.0M.

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28 Certificate of Qualified Person

28.1 William Gilmour, P.Geo.

I, William Gilmour, of Vernon, BC, as a co-author of the *"Spanish Mountain Gold Project Prefeasibility Study NI 43-101 Technical, British Columbia, Canada*" (the Technical Report), with an effective date of May 10, 2021, prepared for Spanish Mountain Gold Ltd., and dated May 31, 2021, do hereby certify that:

- I am a partner in Discovery Consultants with a business address at 10 – 100 Kalamalka Lake Road, Vernon, BC, V1T 7M2.
- I am a graduate of the University of British Columbia, in 1970, with a BSc in Geology.
- I am a member in good standing of the Engineers and Geoscientists BC (Licence # 19743).
- I have practised my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to the Spanish Mountain Gold Project includes the monitoring of analytical drill results since 2011.
- I am responsible for the preparation of Section(s) 4 through 12, 23 and portions of Section 1, 25 and 26 of the technical report related to Geology.
- I last visited the property on December 2, 2020.
- I have had prior involvement with the subject property having co-authored a previous technical report titled "Spanish Mountain Gold, NI 43-1010 Technical Report, based on 2019 Preliminary Economic Assessment" for Spanish Mountain Gold Ltd, December 2, 2019.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have been involved with the Spanish Mountain Gold Project during the preparation of previous Technical Reports.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 31st day of May, 2021 at Vernon, BC.

"Original Signed and Sealed"

William Gilmour, P.Geo.

28.2 Marc Jutras, P.Eng.

I, Marc Jutras, of North Vancouver, British Columbia, as a co-author of the *"Spanish Mountain Gold Project Prefeasibility Study NI 43-101 Technical, British Columbia, Canada*" (the Technical Report), with an effective date of 10 May, 2021, prepared for Spanish Mountain Gold Ltd., and dated May 31, 2021, do hereby certify that:

- I am a Principal, Mineral Resources with Ginto Consulting Inc. with a business address at 333 West 17th Street, North Vancouver, British Columbia.
- I am a graduate of the University of Quebec in Chicoutimi in 1983, and hold a Bachelor's degree in Geological Engineering. I am also a graduate of the Ecole Polytechnique of Montreal in 1989, and hold a Master's degree of Applied Sciences in Geostatistics;
- I am a member in good standing of the Engineers and Geoscientists of British Columbia (License # 24598), the Engineers and Geoscientists of Newfoundland and Labrador (license # 09029), and the Quebec Order of Engineers (license # 38380).
- I have practiced my profession continuously since graduation in 1983. I have worked continuously in the field of mineral resource estimation of numerous international exploration projects and mining operations. I have been involved in the evaluation of mineral resources of precious and base metals at various levels: early to advanced exploration projects, preliminary studies, preliminary economic assessments, prefeasibility studies, feasibility studies and technical due diligence reviews
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 Standards of disclosure for Mineral Projects (NI 43-101).
- I am responsible for the preparation of Sections 12 and 14, and portions of Sections 1, 25, 26, and 27 of the technical report related to the Mineral Resource Estimate.
- I have visited the property on November 25, 2020.
- I have had no prior involvement with the subject property.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have not been involved with the Spanish Mountain Gold Project of previous Technical Reports.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 31st day of May, 2021 at North Vancouver, British Columbia.

"Original Signed and Sealed"

Marc Jutras, P.Eng., M.A.Sc.
Principal, Mineral Resources
Ginto Consulting Inc.

28.3 Marc Schulte, P.Eng.

I, Marc Schulte, P.Eng., as a co-author of the *"Spanish Mountain Gold Project Prefeasibility Study NI 43-101 Technical, British Columbia, Canada*" (the Technical Report), with an effective date of 10 May, 2021, prepared for Spanish Mountain Gold Ltd., and dated 31 May 2021, do hereby certify that:

- I am a Mining Engineer with Moose Mountain Technical Services, with a business address at #210 1510 2nd Street North Cranbrook, BC V1C 3L2.
- I am a graduate of the University of Alberta in 2002 with a Bachelor of Science in Mining Engineering.
- I am a member in good standing of the self-regulating Association of Professional Engineers, Geologist and Geophysicists of Alberta. (#71051)
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience includes working in geology, mine planning and financial modeling for mine operations and mineral project evaluations for precious metals and base metals utilizing open pit mining methods.
- I am responsible for the preparation of Section(s) 15, 16, 19, 22 and portions of Section 1, 21, 25, 26 and 27 of the technical report related to Mineral Reserves, Mining, Market Studies and Contracts, and Economic Analysis.
- I have visited the property on 12 September 2019.
- I have previously co-authored the following Technical Reports on the Spanish Mountain Gold Project:
    - Galbraith, L., Gilmour, W., Giroux, G.H., Meintjes, T., and Schulte, M., 2017: Preliminary Economic Assessment for the Spanish Mountain Gold Property, Likely, British Columbia, Canada: report prepared by Moose Mountain Technical Services, Discovery Consultants, Giroux Consultants, and Knight Piésold Ltd. for Spanish Mountain Gold Ltd., effective date May 17, 2017; amended date 11 February 2019.
    - Bird, S., Galbraith, L., Gilmour, Meintjes, T., and Schulte, M., 2019: Spanish Mountain Gold NI 43-101 Technical Report based on 2019 Preliminary Economic Assessment, Likely, British Columbia, Canada: report prepared by Moose Mountain Technical Services, Discovery Consultants, and Knight Piésold Ltd. for Spanish Mountain Gold Ltd., effective date 2 December 2019.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.

- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 31st day of May 2021

"Original Signed and Sealed"

Marc Schulte, P.Eng.

28.4 Frank Grills, P.Eng.

I, Frank Grills, of North Vancouver, BC, as a co-author of the *"Spanish Mountain Gold Project Prefeasibility Study NI 43-101 Technical, British Columbia, Canada*" (the Technical Report), with an effective date of 10 May, 2021, prepared for Spanish Mountain Gold Ltd., and dated May 31, 2021, do hereby certify that:

- I am the Principal Project Manager with Moose Mountain Technical Services, with a business address at 1975 1st Avenue S Cranbrook, BC, V1C 6Y3.
- I am a graduate of University of Surrey (B. Sc. (Hons), 1979 and the University of Witwatersrand (M.Sc. (Eng), 1985).
- I am a member in good standing of the Engineers and Geoscientists of British Columbia (Registration No. 29591).
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to studies include 34 years' experience in engineering, estimating, construction management and project management of major process plants and studies. Recent studies I have worked on include Premier & Red Mountain Gold Project, Ixtaca Project and Kerr Sulphurets Mitchell Project.
- I am responsible for the preparation of Section(s) 2, 3 and 24, and portions of Section 1, 25, and 26 of the technical report related to overall compilation of the report.
- I have not visited the property.
- I have no prior involvement with the property that is the subject of the Technical Report.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have not been involved with the Spanish Mountain Gold Project during the preparation of previous Technical Reports.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 31st day of May, 2021 at North Vancouver, BC.

"Original Signed and Sealed"

Frank Grills, P.Eng.
Project Manager
Moose Mountain Technical Services

28.5 Malcolm Cameron, P.Eng.

I, Malcolm Cameron, of Lillooet, BC, as a co-author of the *"Spanish Mountain Gold Project Prefeasibility Study NI 43-101 Technical, British Columbia, Canada*" (the Technical Report), with an effective date of 10 May, 2021, prepared for Spanish Mountain Gold Ltd., and dated May 31, 2021, do hereby certify that:

- I am President of MCA Engineering Ltd. with a business address at 917 Columbia St, Lillooet, BC, V0K-1V0.
- I am a graduate of UBC, BASc, 1976.
- I am a member in good standing of the Engineers and Geoscientists of British Columbia 11909.
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to Electrical Power Systems includes 40 years industrial experience.
- I am responsible for the preparation of Section(s) 18.4, 18.5, 21.1.9 and portions of Section 1, 21, 25, and 26 of the technical report related to Off-Site Electrical Power Transmission and on-site electrical receiving substation.
- I have not visited the property.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have been involved with the Spanish Mountain Gold Project during the preparation of previous Technical Reports.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 31st day of May, 2021 at Lillooet, BC.

"Original Signed and Sealed"

Malcolm Cameron, P.Eng.
President
MCA Engineering Ltd.

28.6 Paul Staples, P.Eng.

I, Paul Staples, of Kaleden, BC as a co-author of the *"Spanish Mountain Gold Project Prefeasibility Study NI 43-101 Technical, British Columbia, Canada*" (the Technical Report), with an effective date of 10 May, 2021, prepared for Spanish Mountain Gold Ltd., and dated May 31, 2021, do hereby certify that:

- I am VP and Global Practice Lead with Ausenco Engineering Canada with a business address at 855 Homer Street, Vancouver, BC.
- I am a graduate of Queen's University in 1993 with a Bachelor of Science degree in Materials and Metallurgical Engineering.
- I am a member in good standing of the Engineers and Geoscientists of British Columbia, License #47467.
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101 for those sections of the Technical Report which I am responsible for preparing.
- My relevant experience includes process operations, design and management of over 50 similar studies or projects in Canada and overseas.
- I am responsible for the preparation of Sections 13, and 17 and portions of Section 18, 21, 25, 26, and 27 of the technical report related to Mineral Processing, Process Plant and Non-process Related Infrastructure.
- I visited the property on Dec. 8, 2020.
- I have had no prior involvement with the subject property.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- As of the date of this Certificate, to my knowledge, information, and belief, the sections of the technical report for which I am responsible contains all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Signed and dated this 31st day of May, 2021 at Kaleden, BC.

"Original Signed and Sealed"

L. Paul Staples, P.Eng.

28.7 Les Galbraith, P.Eng.

I, Les Galbraith, of Vancouver, British Columbia, as a co-author of the *"Spanish Mountain Gold Project Prefeasibility Study NI 43-101 Technical, British Columbia, Canada*" (the Technical Report), with an effective date of 10 May, 2021, prepared for Spanish Mountain Gold Ltd., and dated May 31, 2021, do hereby certify that:

- I am a Specialist Engineer | Associate with Knight Piésold Ltd. with a business address at Suite 1400 – 750 West Pender Street, Vancouver. B.C. V6C 2T8.
- I am a graduate of the University of British Columbia, B.A.Sc. (Civil Engineering) 1995.
- I am a member in good standing of the Engineers and Geoscientists of British Columbia (#25493).
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- I have over 25 years of relevant experience in providing civil and geotechnical engineering support to mining projects.
- I am responsible for the preparation of Sections 18.3.1; 18.3.2; 18.3.3; 18.3.4; 18.3.5; 18.3.6; 18.3.7; 18.3.8, 21.1.11; 21.1.12; 21.2.3; 21.3.15, Section 20 (with the exception of 20.3), and portions of Section 1, 25, 26, and 27 of the technical report related to the Tailing Storage Facility and Water Management, and Environmental Studies, Permitting, and Social or Community Impact.
- My most recent site visit was on September 12, 2019.
- I have had prior involvement with the subject property having co-authored a previous technical report titled "Spanish Mountain Gold, NI 43-101 Technical Report based on 2019 Preliminary Economic Assessment" prepared by Spanish Mountain Gold Ltd., in December 2019.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have been involved with the Spanish Mountain Gold Project during the preparation of previous Technical Reports.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 31st day of May, 2021 at Vancouver, British Columbia.

"Original Signed and Sealed"

Les Galbraith, P. Eng.
Specialist Engineer | Associate

28.8 Sam Billin, P.Eng.

I, Sam Billin, of Elko, Nevada, as a co-author of the *"Spanish Mountain Gold Project Prefeasibility Study NI 43-101 Technical, British Columbia, Canada*" (the Technical Report), with an effective date of 10 May, 2021, prepared for Spanish Mountain Gold Ltd., and dated May 31, 2021, do hereby certify that:

- I am a Founder and President with a business address at 2720 Ruby Vista Drive, Suite 101, Elko, NV 89801.
- I am a graduate of Brigham Young University (Civil Engineering, 1991).
- I am a member in good standing of the Engineers and Geoscientists of British Columbia (License #182952).
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to water treatment includes 30 years of specialization in the scoping, design, construction, and operation of water treatment plants for the mining industry.
- I am responsible for the preparation of Section(s) 18.3.19, and portions of Section 21, 25, 26, and 27 of the technical report related to the Water Treatment Facilities.
- I have not visited the property.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 31st day of May, 2021 at Elko, Nevada.

"Original Signed and Sealed"

Samuel Billin, Water Treatment Expert
President
Linkan Engineering

28.9 Andrea Samuels, P.Geo.

I, Andrea Samuels, of Maple Ridge, B.C., as a co-author of the *"Spanish Mountain Gold Project Prefeasibility Study NI 43-101 Technical, British Columbia, Canada*" (the Technical Report), with an effective date of 10 May, 2021, prepared for Spanish Mountain Gold Ltd., and dated May 31, 2021, do hereby certify that:

- I am a Senior Geochemist with pHase Geochemistry Inc. with a business address at 1032 Keith Rd W, North Vancouver, BC, Canada, V7P 1Y5
- I am a graduate of Queen's University in 1997 with a Bachelor of Science (Honours) degree in Geological Sciences.
- I am a member in good standing of the Engineers and Geoscientists of British Columbia (#38156).
- I have practiced my profession for 20 years since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to the technical report includes 16 years experience working as a geochemist in the consulting industry specializing in the assessment of acid rock drainage/metal leaching (ARD/ML) potential and source term predictions of waste rock and tailings for various exploration and mining projects in Canada, the United States, South America, and Australia.
- I am responsible for the preparation of Section(s) 1.25, 1.26, 20.3, 25.5, and 26.4 of the technical report related to Geochemistry.
- I have not visited the Spanish Mountain Gold Project.
- I have had no prior involvement with the property that is the subject of the technical report.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have not been involved with the Spanish Mountain Gold Project during the preparation of previous Technical Reports.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- As of the date of this Certificate, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 31st day of May, 2021 at Maple Ridge, B.C.

"Original Signed and Sealed"

Andrea Samuels, P.Geo.
Senior Geochemist / pHase Geochemistry Inc.

29 Date and Signature Page

The undersigned prepared this Technical Report, titled "*Spanish Mountain Gold Project Prefeasibility Study NI 43-101 Technical Report, British Columbia,"* and dated May 31, 2021, in support of the public disclosure for public listing. The format and content of this report conforms to National Instrument 43-101 (NI 43-101) of the Canadian Securities Administrators.

"Original Signed and Sealed"

William Gilmour, P.Geo.
Geologist
Discovery Consultants

"Original Signed and Sealed"

Marc Jutras, P.Eng.
Principal, Mineral Resources
Ginto

"Original Signed and Sealed"

Marc Schulte, P.Eng.
Mining
Moose Mountain Technical Services

"Original Signed and Sealed"

Frank Grills, P.Eng.
Principal Project Management
Moose Mountain Technical Services

"Original Signed and Sealed"

Malcolm Cameron P.Eng.
Electrical Engineer
MCA Engineering Ltd.

"Original Signed and Sealed"

Paul Staples, P.Eng.
VP & Global Practice Lead, Comminution
Ausenco Canada Inc.

"Original Signed and Sealed"

Les Galbraith, P.Eng.
Specialist
Knight Piesold

"Original Signed and Sealed"

Sam Billin, P.Eng.
President/Founder
Linkan Engineering

"Original Signed and Sealed"

Andrea Samuels, P.Geo.
Geochemist
pHase Geochemistry Inc.