Spanish Mountain Gold NI 43-101 Technical Report based on 2019 Preliminary Economic Assessment

Likely, British Columbia, Canada

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

Effective Date for Mineral Resources: October 10, 2019
Date of Technical Report: December 2, 2019

Report Authors:

Marc Schulte, P.Eng.
William Gilmour, P.Geo.
Sue Bird, P.Eng.
Les Galbraith, P.Eng.
Tracey Meintjes, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Marc Schulte, P.Eng., am employed as a Mining Engineer with Moose Mountain Technical Services, with an office address of #210 1510 2nd Street North Cranbrook, BC V1C 3L2.

This certificate applies to the technical report titled “Spanish Mountain Gold, NI 43-101 Technical Report, based on 2019 Preliminary Economic Assessment” that has an effective date of December 2, 2019 (the “technical report”).

I am a member of the self-regulating Association of Professional Engineers, Geologist and Geophysicists of Alberta. (#71051). I graduated with a Bachelor of Science in Mining Engineering from the University of Alberta in 2002.

I have worked as a Mining Engineer for a total of 17 years since my graduation from university. I have worked on precious metals, base metals and coal mining projects, including mine operations and evaluations.

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).

My most recent site visit was on September 12, 2019.

I am responsible for Sections 1, 2, 3, 15, 16, 18, 19, 21, 22, 24, 27 and portions of Sections 1, 25 and 26 of the technical report related to Mine Operations and Infrastructure Planning.

I am independent of Spanish Mountain Gold Ltd. as independence is described by Section 1.5 of NI 43–101.

I have previously co-authored the following 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 February 11, 2019.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated: December 2, 2019

{Signed and Sealed}

____

Signature of Qualified Person
Marc Schulte, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, William Gilmour, P.Geo., am a Geologist with Discovery Consultants, with an office address of 101-2913, 29th Avenue, Vernon, BC, V1T 1Z2.

This certificate applies to the technical report titled "Spanish Mountain Gold, NI 43-101 Technical Report, based on 2019 Preliminary Economic Assessment" that has an effective date of December 2, 2019 (the "technical report").

I am a member of the self-regulating Association of Professional Engineers and GeoScientists of British Columbia. (#19743). I graduated with a Bachelor of Science in Geology from the University of British Columbia in 1970.

I have been practicing my profession since graduation from university. I have over 45 years of experience in mineral exploration for a variety of base and precious metals. My working experience includes grassroots and reconnaissance exploration, project evaluation, geological mapping, planning and execution of drill programs, and project reporting.

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).

My most recent site visit was on September 12, 2019.

I am responsible for Sections 4 through 12, 23 and portions of Section 1, 25 and 26 of the technical report related to Geology.

I am independent of Spanish Mountain Gold Ltd. as independence is described by Section 1.5 of NI 43–101.

I have previously co-authored the following 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 February 11, 2019.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated: December 2, 2019

{Signed and Sealed}

____
Signature of Qualified Person
William Gilmour, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

I, Sue Bird, P.Eng., am employed as a Geological Engineer with Moose Mountain Technical Services, with an office address of #210 1510 2nd Street North Cranbrook, BC V1C 3L2.

This certificate applies to the technical report titled "Spanish Mountain Gold, NI 43-101 Technical Report, based on 2019 Preliminary Economic Assessment" that has an effective date of December 2, 2019 (the "technical report").

I am a member of the self-regulating Association of Professional Engineers and GeoScientists of British Columbia. (#25007). I graduated with a Geologic Engineering degree (B.Sc.) from the Queen's University in 1989 and a M.Sc. in Mining from Queen's University in 1993.

I have worked as an engineering geologist for over 25 years since my graduation from university. I have worked on precious metals, base metals and coal mining projects, including mine operations and evaluations.

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 visited the property on September 12, 2019.

I am responsible for Section 14 and portions of Sections 1, 25 and 26 of the technical report related to the Mineral Resource.

I am independent of Spanish Mountain Gold Ltd. as independence is described by Section 1.5 of NI 43–101.

I have not previously co-authored reports on the Spanish Mountain Gold Project.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated: December 2, 2019

{Signed and Sealed}

____

Signature of Qualified Person
Sue Bird, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Les Galbraith, P.Eng., am employed as a Specialist Engineer / Project Manager with Knight Piésold Ltd., with an office address of Suite 1400 – 750 West Pender Street, Vancouver, BC, V6C 2T8.

This certificate applies to the technical report titled “Spanish Mountain Gold, NI 43-101 Technical Report, based on 2019 Preliminary Economic Assessment” that has an effective date of December 2, 2019 (the “technical report”).

I am a member of the self-regulating Association of Professional Engineers and GeoScientists of British Columbia. (#25493). I graduated with a Bachelor of Science from the University of British Columbia in 1995.

I have 22 years of relevant experience in providing civil and geotechnical engineering support to mining and hydroelectric project.

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).

My most recent site visit was on September 12, 2019.

I am responsible for Sections 18.4, 20 and portions of Sections 1, 25 and 26 of the technical report related to the Tailings Storage Facility.

I am independent of Spanish Mountain Gold Ltd. as independence is described by Section 1.5 of NI 43–101.

I have previously co-authored the following 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 February 11, 2019.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated: December 2, 2019

{Signed and Sealed}

____

Signature of Qualified Person
Les Galbraith, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Tracey Meintjes, P.Eng., am employed as a Metallurgical Engineer with Moose Mountain Technical Services, with an office address of #210 1510 2nd Street North Cranbrook, BC V1C 3L2.

This certificate applies to the technical report titled "Spanish Mountain Gold, NI 43-101 Technical Report, based on 2019 Preliminary Economic Assessment" that has an effective date of December 2, 2019 (the "technical report").

I am a member of the self-regulating Association of Professional Engineers and GeoScientists of British Columbia. (#37018). I am a graduate of Technikon Witwatersrand (NHD Extraction Metallurgy – 1996).

My relevant experience includes process engineering and supervision in South Africa and North America. My precious metals project experience includes both operations and metallurgical process development. I have been working in my profession continuously since 1996.

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 have not visited the Property

I am responsible for Sections 13 and 17 and portions of Sections 1, 25 and 26 of the technical report related to Metallurgy and Processing.

I am independent of Spanish Mountain Gold Ltd. as independence is described by Section 1.5 of NI 43–101.

I have previously co-authored the following 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 February 11, 2019.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated: December 2, 2019

{Signed and Sealed}

____

Signature of Qualified Person
Tracey Meintjes, P.Eng.

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

1.0 Summary

1.1 Introduction

Marc Schulte, P.Eng. William Gilmour, P.Geo., Sue Bird, P.Eng., Les Galbraith, P.Eng., and Mr. Tracey Meintjes, P.Eng., have prepared an NI 43-101 Technical Report (the Report) on the Spanish Mountain Gold Project (the Project) for Spanish Mountain Gold Ltd (Spanish Mountain). The Report is based on an updated Mineral Resource Estimate and Preliminary Economic Assessment (PEA) on the Project.

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 (Figure 1-1).

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 3,650,000 t/a (or 10,000 t/d) of gold and silver bearing material from an open pit operation and will produce gold-silver doré as a final product.

1.2 Terms of Reference

The Report has been prepared in support of disclosures in Spanish Mountain's news release dated 23 October 2019, entitled "Spanish Mountain Gold Announces Results of Preliminary Economic Assessment for Phase 1 Project".

An updated Mineral Resource estimate and PEA has been completed for the Project in 2019. Information from this study has been summarized into this Report in the relevant sections.

The preliminary economic assessment is preliminary in nature; the preliminary economic assessment is based on resources, not reserves. Resources are considered too speculative geologically to have economic considerations applied to them, so the project does not yet have proven economic viability.

All currency amounts are referred to in Canadian dollars ($ or C$) unless otherwise indicated. All measurements are in metric units unless otherwise indicated. Figures throughout the Report are plotted on UTM coordinate system WGS 84 Zone 10U.

General information for the Project is summarized in Table 1-1.

.

Table 1-1 General PEA Results

Description	Unit	Amount
Mill Feed Production	Mt	39
Average Mill Feed Grade	g/t Au	1.00
Life-of-mine (LOM)	years	11
Milling Rate	t/d	10,000
Strip Ratio	t/t	3.5
Total Project Initial Capital Cost	C$, millions	364
Average Overall Operating Cost	C$/t milled	19.10
Gold Price	US$/oz	1,275
Pre-Tax Net Present Value (NPV) at 5% Discount Rate	C$, millions	414
Pre-Tax Internal Rate of Return (IRR)	%	23
After-Tax Net Present Value (NPV) at 5% Discount Rate	C$, millions	325
After-Tax Internal Rate of Return (IRR)	%	21
Capital Payback Period	years	3.5

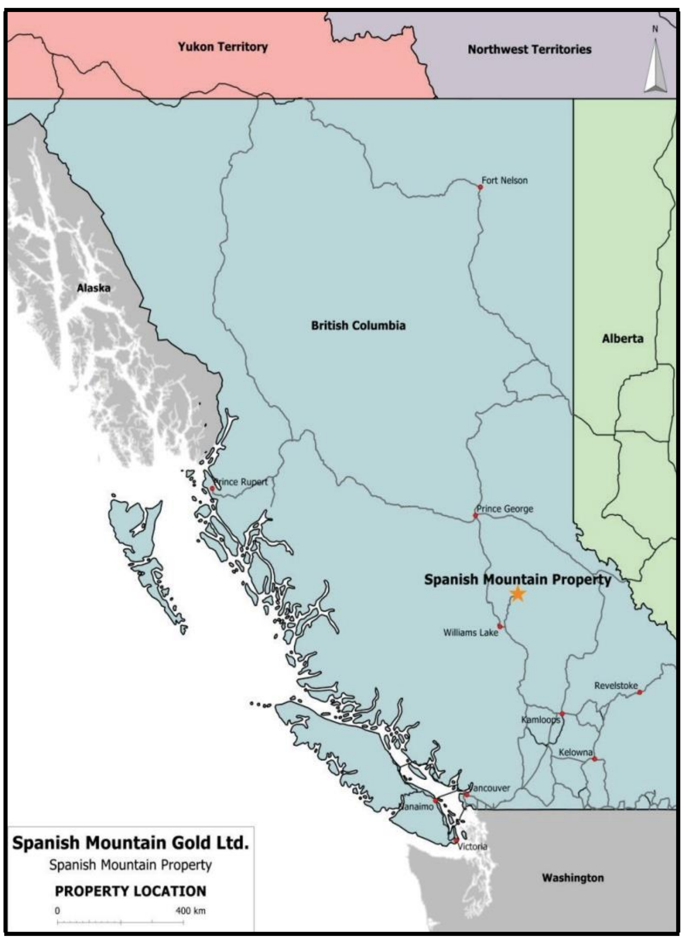


Figure 1-1 Property Location Map

1.3 Property Description

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.

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).

1.4 Geological Setting

Geologically, the Property lies within the central part of the Quesnel Terrane, which around the Property consists of a sedimentary package of black, graphitic argillites, phyllitic siltstones, sandstones, limestones and banded tuffs of the Late Triassic Nicola Group. The sedimentary rocks have been metamorphosed to sub-greenschist grade and are locally intruded by plagioclase-quartz-hornblende sills and dykes.

The Spanish Mountain gold deposit is 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.

1.5 Mineral Resource Estimate

Mineral Resources are reported using the 2014 CIM Definition Standards (CIM, 2014). The Qualified Person for the Resource Estimate is Sue Bird, P.Eng., who is independent of Spanish Mountain Gold.

The base case Mineral Resource Estimate at a 0.15 g/t Au cut-off is summarized in Table 1-2. The sensitivity to cutoff grade is summarized in Section 14 of the Report. The resource has been confined to a “reasonable prospects of eventual economic extraction” shape, based on conventional open pit mining with the following assumptions used to determine the cutoff grade:

- Gold Price = US$1,275/oz;
- Exchange Rate = 0.75 US$:1 C$;
- Process Costs (including G&A costs) = $7.25/t;
- Process Recovery = 90%; and
- Overall Slope Angles conforming into inputs listed in Table 16-5.

Table 1-2 Mineral Resource Estimate within Constraining Pit

Classification	Tonnage	Grade		Contained Metal	
	Mt	Au, g/t	Ag, g/t	Au, koz.	Ag, koz.
Measured	29.6	0.60	0.83	569	791
Indicated	243.6	0.46	0.69	3,566	5,413
Measured + Indicated	**273.2**	**0.47**	**0.71**	**4,135**	**6,204**
Inferred	52.4	0.37	0.67	619	1,128

Notes for Resource Tables:

- *Mineral Resources have an effective date of October 10, 2019 and are prepared in accordance with CIM Definition Standards and NI 43-101. The Qualified Person for the estimate is Sue Bird, P.Eng.*
- *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.*
- *Inferred Resources are not based on sufficient drilling to be considered Measured or Indicated and it is not certain that further exploration will result in upgrading the classification. As such, Inferred resources have not been used in the mine plan.*
- *Silver value is not considered in the cut-off grade estimation.*
- *Considerations for the Lerchs-Grossman algorithm used to define the "reasonable prospects of eventual economic extraction" open pit shell are the same as those listed above for the cutoff grade determination, as well as a $2.20/t mining cost. Overall pit slope angles range from 20 degrees to 43 degrees and are estimated based on geotechnical analysis of various zones in the deposit.*

Factors that may affect the estimates include: metal price assumptions, changes in interpretations of mineralization geometry and continuity of mineralization zones, changes to kriging assumptions, metallurgical recovery assumptions, operating cost assumptions, confidence in the modifying factors, including assumptions that surface rights to allow mining infrastructure to be constructed will be forthcoming, delays or other issues in reaching agreements with local or regulatory authorities and stakeholders, and changes in land tenure requirements or in permitting requirement.

1.6 Metallurgical Testing and Mineral Processing

Sample material representative of the major rock types present in the Spanish Mountain Gold deposit have been used in a series of metallurgical test programs including comminution, gravity concentration, multistage flotation, flotation concentrate regrind and cleaning, cyanide leach tests and cyanide destruction tests.

Coarse grinding to a P_{80} of 180 µm of a moderate to soft mill feed is required for rougher flotation. Flotation concentrate cleaning using Carboxymethylcellulose (CMC) is utilized to depress organic carbon from flotation concentrate. Gravity concentration is used to scavenge gold from the cleaner tails.

A combined leach feed concentrate represents a mass pull that is 3.4% of total mill feed.

Concentrate regrind to a P_{80} of 35 µm is required prior to leaching. Gold is predominantly associated with quartz and sulphide (mainly pyrite) minerals.

The 10,000 t/d process plant flowsheet design includes crushing, grinding, multistage flotation, scavenger gravity concentration of cleaner tails, concentrate regrind, and CIL to produce doré. CIL tailings is pumped to cyanide destruction using the SO_2/Air process, where test work supports cyanide levels reduced to acceptable environmental levels prior to disposal to the tailings storage facility (TSF).

A gold flotation plus gravity scavenging of the cleaner/recleaner tails recovery of 92% and a gold Carbon in Leach (CIL) recovery of 99% results in an overall gold recovery of 91% with a low cyanide consumption of 0.1 kg/t ore. Overall silver recovery is an estimated 27%.

1.7 Mining

The Spanish Mountain deposit will be mined using a conventional open pit mining method, using off-highway haul trucks and hydraulic shovels. The waste and mineralized rock will be drilled and blasted, with separation identified using bench scale grade control methods.

A PEA level mine operation design, approximately 11-year open pit production schedule, and mining cost model have been developed. The potential in-pit tonnages, based on a 0.40 g/t gold cut-off, are summarized in Table 1-3. These quantities and grades are a subset of the mineral resources described in Section 14.0 of this Technical Report. The mine production schedule is described in Figure 1-2.

Table 1-3 Mining ROM Production

	Amount	Unit
Measured and Indicated Mill Feed	39,097	kt
Gold Grade	1.00	g/t
Gold Contained	1,258	koz.
Silver Grade	0.74	g/t
Silver Contained	927	koz.
Waste	138,541	kt
Strip Ratio	3.5	t/t

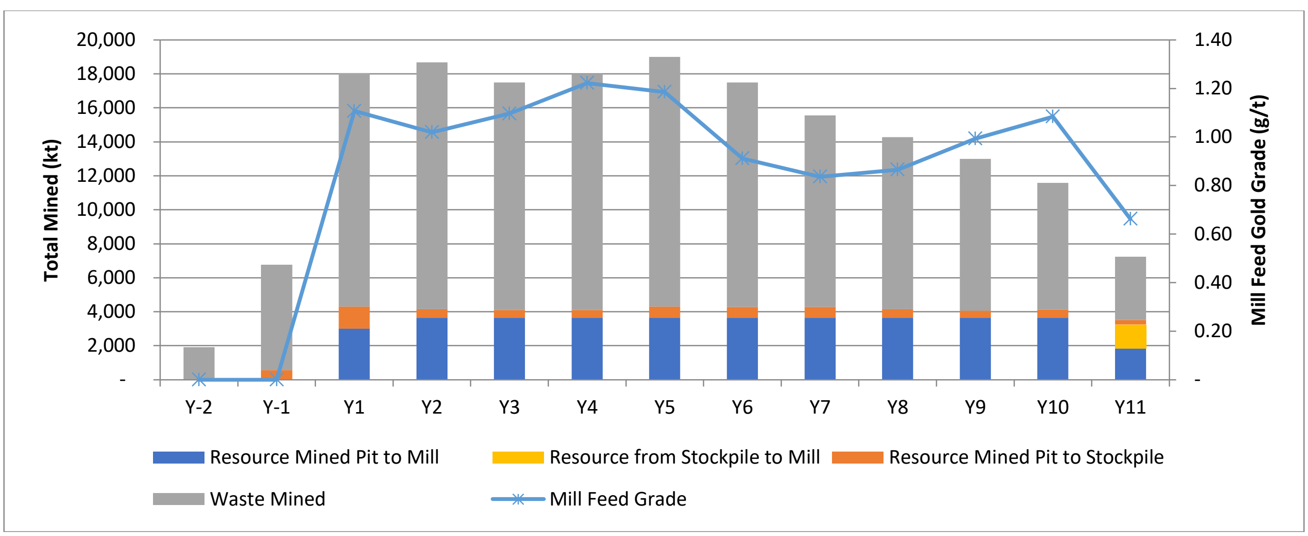


Figure 1-2 Mine Production Schedule

1.8 Project Infrastructure

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 FSR. This road currently travels through the proposed mine site; it will require rerouting to accommodate the location of the north WRSF and open pit. Access to this FSR route through the site will be maintained throughout the LOM.

On-site infrastructure includes:

- Electrical Substation and distribution
- Process Plant
- Tailing Storage Facility
- Water Storage Pond
- Maintenance and Truck Shop
- Administration/Dry Building
- Assay Laboratory
- Cold Storage Warehouse
- Access roads
- Water Supply
- Wastewater treatment systems
- Solid waste disposal facilities and sewage plant
- Communication systems
- Medical facilities
- Site support systems including workshops, maintenance shop, warehousing and security

The Project requires approximately 10.5 MW of peak load for 10,000 t/d operation demand. The power will be supplied by a new transmission line interconnecting the SMG site to BC Hydro’s power system. A new 230 kV transmission line directly from a new BC Hydro 230 kV switching station adjacent to BC Hydro’s existing 500 kV McLeese Capacitor station to the SMG site is the base case for the external power supply.

A layout of all important features, including the open pit, rock stockpiles, haul rocks and on-site infrastructure is shown in Figure 1-3. The deposit itself is centred on the open pit. The mineral claim boundaries are shown on this Figure for reference.

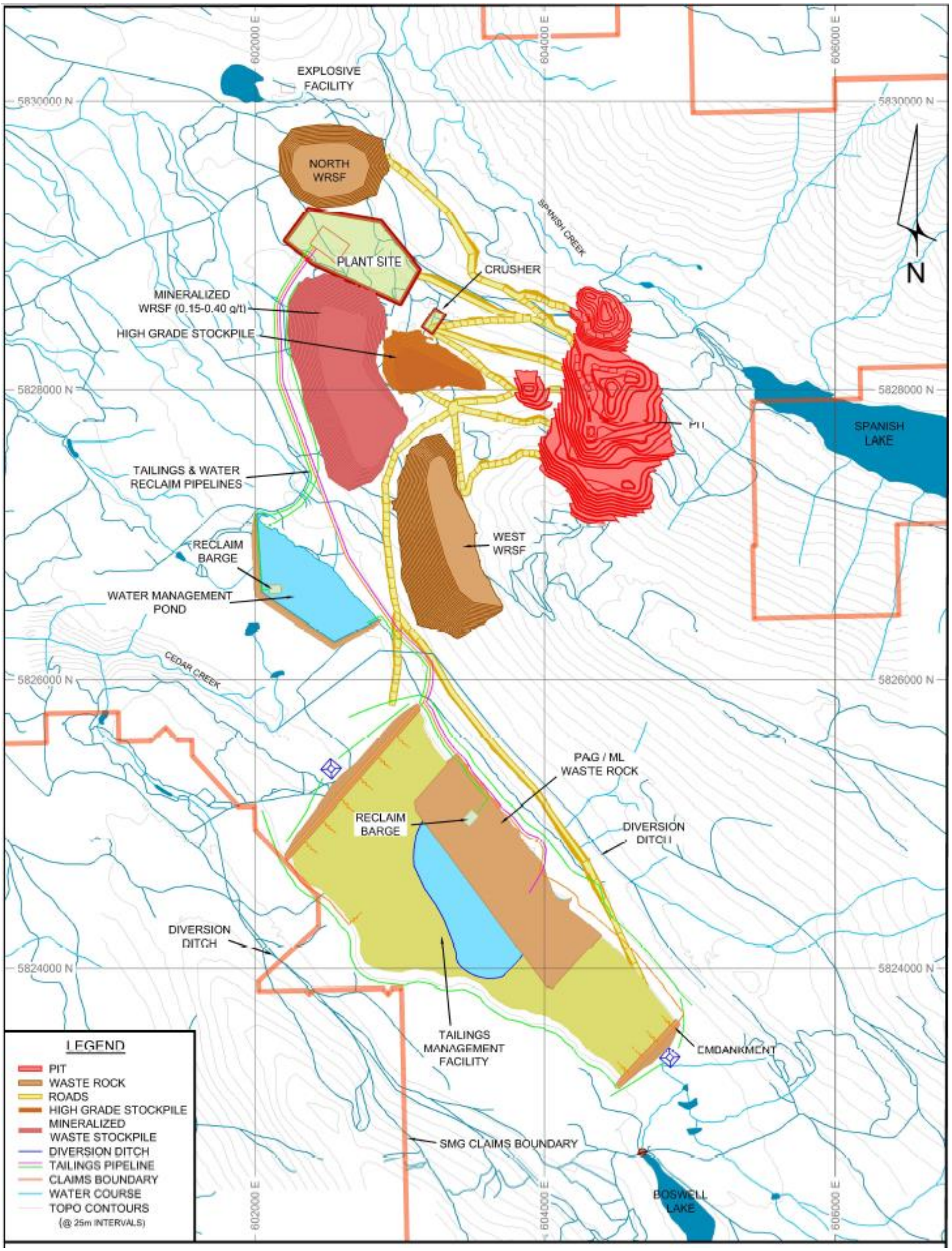


Figure 1-3 General Arrangement Layout

1.9 Waste and 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 processing plant will produce two tailings streams: rougher tailings and cleaner/CIL tailings, which will be transported from the plant site to the TSF in separate pipelines. Each tailings stream will be deposited independently; the rougher tailings will be discharged along the TSF embankments to create tailings beaches and the cleaner tailings will be discharged subaqueously in the supernatant pond and progressively encapsulated by the rougher tailings.

The TSF capacity at all stages of the mine life includes the supernatant pond volume and allowances for wave run-up, post-seismic settlement, sloping beaches and containment of the inflow design flood. The final capacity of the TSF will be approximately 39 Mt of tailings, 25 Mt of PAG/ML waste rock, plus the supernatant pond volume and freeboard allowances.

Water will generally be directed to and stored in the water management pond, and not within the TSF.

The TSF embankments will be constructed using suitable waste rock and overburden (low permeability glacial till) from the open pit.

1.10 Environmental and Social License

Environmental studies—including studies on surface and groundwater quality and quantity, geochemistry, climatology, fish and fish habitat, wildlife, and vegetation—were initiated in 2007 at the Project site and continued through 2011.

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 is expected to have a material impact on the ability to extract the mineral resources or reserves.

The Project will require approval under the federal and provincial environmental assessment (EA) process prior to receiving the necessary permits and authorizations for construction and operation. The federal Fisheries Act prohibits the harmful alteration, disruption, or destruction of fish habitat without specific authorization.

Construction of the TSF in the Nina Lake basin of the Cedar Creek watershed may require a Schedule 2 Amendment under the Metal Mining Effluent Regulations (MMER) of the Fisheries Act. Fish habitat compensation will be required to balance the loss of any habitat resulting from construction and operation of the Project.

Public comment in relation to the Project must be sought, addressed, and documented through public open houses, meetings and presentations, and through the provincial and federal EA registries.

The Project is situated within the asserted traditional territories of the T'exelc (Williams Lake) and Xats'ull/Cmetem' (Soda Creek) First Nations, both of whom are member nations of the Northern Secwepemc te Qelmucw (Northern Shuswap Tribal Society Council), as well as the Lhtako Dene Nation (Red Bluff Indian Band), which is part of the Carrier Chilcotin Tribal Council. SMG has 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.

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 (WRSF's and tailings), ultimate pit wall compositions, hydrologic regimes, and end use. These studies have been initiated and are typically completed as part of a Feasibility Study.

1.11 Capital and Operating Costs

The total estimated pre-production capital cost for the design, construction, and installation and commissioning for all facilities and equipment is shown in Table 1-4.

The accuracy of the estimate is ±40%. This study has been prepared with a base date of Q4 2019 with no provision for escalation. All Capital and Operating costs are reported in Canadian dollars unless specified otherwise; an exchange rate of US$0.75 to C$1.00 has been used for any conversions.

Table 1-4 Capital Cost Summary

Direct Costs	**Initial Capital Cost (M$)**
Overall Site	6.7
Open Pit Mining	70.2
Ore Handling	24.0
Processing Plant (including Ore Handling)	53.4
Tailing Storage Facility & Water Management	46.7
Environmental	12.0
On-Site Infrastructure	24.0
Off-Site Infrastructure	17.1
Direct Costs Sub-Total	**254.1**
Indirect Costs	
Project Indirects	58.9
Owner's Costs	9.3
Contingencies	41.5
Indirect Costs Sub-Total	**109.7**
Total Initial Capital Cost	**363.8**
Total Sustaining Capital Cost	**57.8**

The unit costs summarized in Table 1-5 are based on an annual production rate of 10,000 t/d, and 365 d/a of operation.

Table 1-5 Operating Cost Summary

Area	Unit Cost
Mining ($/t mined)	$2.48
Mining ($/t milled)	$10.73
Processing ($/t milled)	$6.14
Tailings ($/t milled)	$0.16
G&A ($/t milled)	$2.06
Total ($/t milled)	**$19.10**

A summary of the life of mine cash operating and all-in sustaining cost/oz. is set out in the table below (in Canadian funds).

Table 1-6 Life of Mine Cash Operating and All-in Sustaining Costs, C$/oz. Produced

Unit Production Costs per ounce	First 5-Yrs	Life of Mine
Cash Cost	$616	$657
All-in-Sustaining Cost (AISC)	$692	$733
Total Cost	$1,035	$1,075

1.12 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.

The preliminary economic assessment is based on resources, not reserves. Resources are considered too speculative geologically to have economic considerations applied to them, so the project does not yet have proven economic viability.

For the 11-year project life, and 39 Mt resource inventory, the following pre-tax financial parameters were calculated:

- 23% IRR
- 3.5-year payback on $364 million capital
- $414 million NPV at 5% discount value.

The following post-tax financial parameters were calculated:

- 21% IRR
- 3.5-year payback on $364 million capital
- $325 million NPV at 5% discount rate.

The following parameters are used for the financial analysis:

- Gold price of US$1,275/oz.
- Silver price of US$18/oz.

- Exchange rate of US$0.75 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 below:

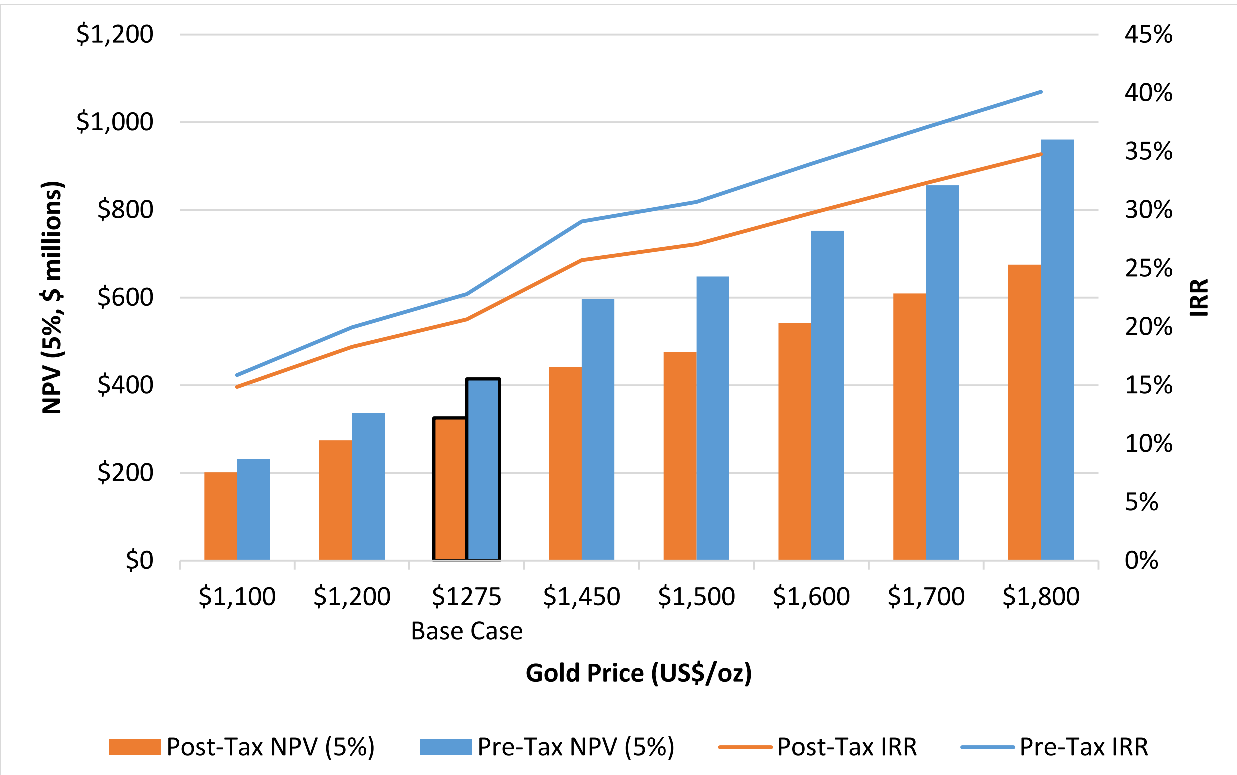


Figure 1-4 Project Economic Sensitivity

1.13 Conclusions and Recommendations

The Project is well suited for open pit mining operations.

The recovery of gold from the Spanish Mountain Gold resource uses conventional processing technology with a relatively coarse primary grind size. High recoveries will be obtained with processing costs being modest, due mainly to their relative low grinding power requirements and low reagent consumption.

The capital costs for the Project have been estimated using local rates and are well within the range found in similar projects.

The Project has a planned 11-year operation for life of mine production of 1.1M ounces of gold at an average mill feed grade of 1.00 g/t.

The Project exhibits positive economics at a range of metal prices.

The positive conclusions of this PEA lead the authors to recommend that the Project should proceed towards a higher level of study.

2.0 Introduction

The purpose of this Technical Report is to present the results of the PEA of Spanish Mountain Gold's mineral resource property located in British Columbia, Canada.

The Technical Report has been prepared by MMTS in conjunction with Discovery and KP 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 drillhole information; as well as scoping level engineering and costing on the open pit, process facilities, site infrastructure and tailings facilities. The authors, in writing this report, used sources of information as listed in the references listed in Section 27.0.

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.

The effective date of the Mineral Resources is October 10, 2019.

The date of the Technical Report is December 2, 2019.

Several authors contributed to or supervised the completion of this Technical Report and are all independent Qualified Persons ("QP") within the meaning of Canadian Securities Administrator's National Instrument 43-101 Standards. Each QP in this report takes responsibility for their work as outlined in the QP Certificates included in this report and found in the following Table:

Table 2-1 QP/Author Responsibility Chart

Qualified Person	Company	Sections of Responsibility
Bill Gilmour, P.Geo.	Discovery	1.3, 1.4, 4 to 12, 23, 25.1, 25.2
Sue Bird, P.Eng.	MMTS	1.5, 14, 25.3, 26.1
Marc Schulte P.Eng.	MMTS	1.1, 1.2, 1.7, 1.8, 1.11, 1.12, 1.13, 2, 3, 15, 16, 18, 19, 21, 22, 24, 25.4, 25.6, 26.2, 26.4 (portions), 27
Tracey Meintjes, P.Eng.	MMTS	1.6, 13, 17, 25.5, 26.3
Les Galbraith, P.Eng.	KP	1.9, 1.10, 18.4, 20, 25.6, 26.4 (portions)

The following lists the latest site visit status of the QP's.

- Bill Gilmour, P.Geo., conducted a site visit on September 12, 2019. During the site visit he inspected some drillhole collars. Time was spent with the geology team discussing sampling methods.
- Sue Bird., P.Eng., conducted a site visit on September 12, 2019. During the site visit she inspected some drillhole collars. Drill core of historic drilled holes with significant intersections

of gold mineralization and visible gold observations were examined. Discussions with field personnel helped geologic modelling and the resource estimation.

- Marc Schulte, P.Eng., conducted a site visit on September 12, 2019. During that site visit he viewed the general topography, inspected proposed pit and stockpile locations, and the locations of existing and proposed infrastructure.
- Tracey Meintjes, P.Eng., has not conducted a site visit.
- Les Galbraith, P.Eng., conducted a site visit on September 12, 2019, to view the general topography and the proposed location of site facilities.

The above site visits were carried out for the Issuer for the purposes of independent review, and the QP's agree in considering the site visits as current for the Issuer as no material scientific or technical information change has occurred between these visits and effective date of this Technical Report.

3.0 Reliance on Other Experts

The QP's have not relied upon any other experts in the preparation of this report.

4.0 Property Description and Location

4.1 Location

The Property is located in the Cariboo region of central British Columbia, approximately six km southeast of Likely and 66 km northeast of Williams Lake (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.

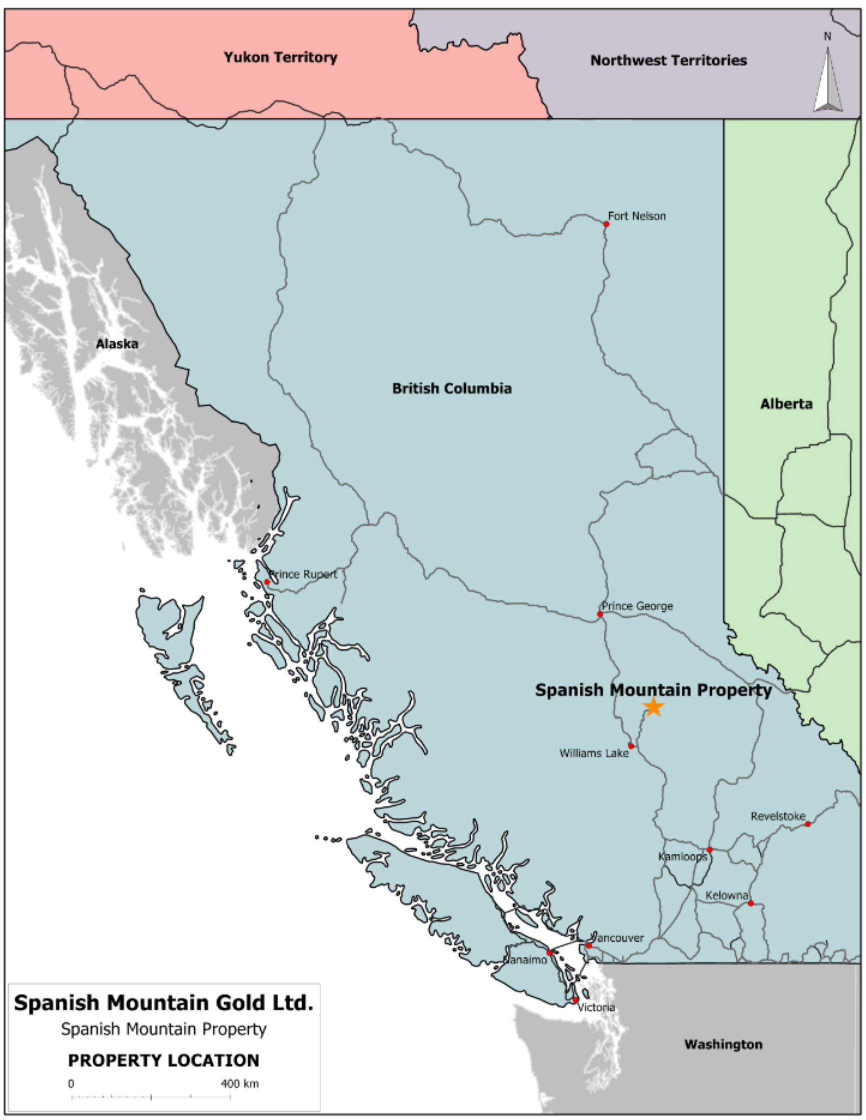


Figure 4-1 Property Location

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 of a third party.

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).

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.

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
502608	SPANISH 2	157.23	093A.053/054	"	2030/Feb/27
503338	SPANISH 3	196.58	093A.053/054	"	2030/Feb/27

Tenure Number	Claim Name	Area (ha)	Map Number	Registered Owner	Good To Date**
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
** Good to Date is dependent on the acceptance of this report
Δ Claim 822682 is converted from legacy claim 204727, which is subject to the Mickle option agreement

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. 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, as the case may be, 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 with 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.

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. 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.

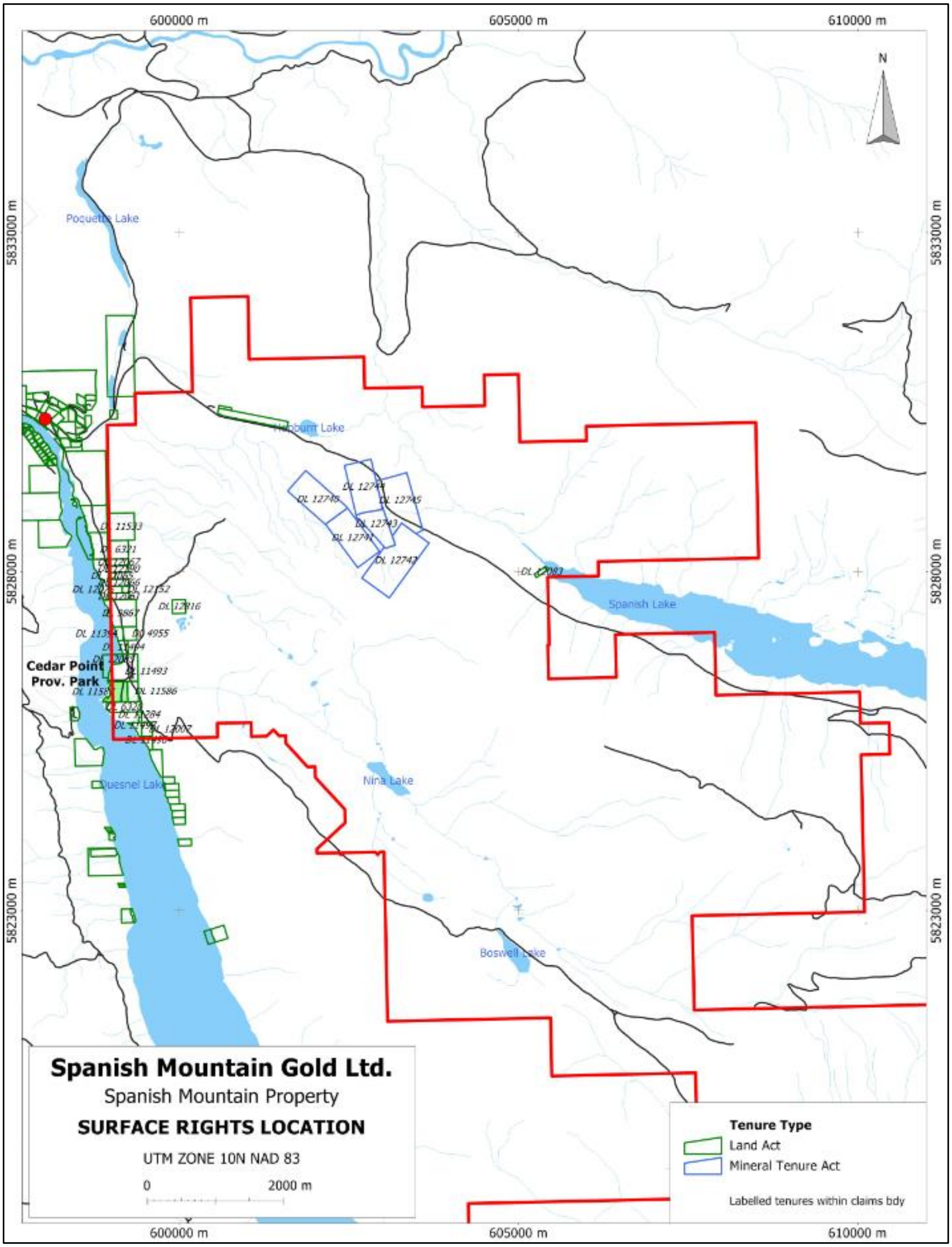


Figure 4-2 Surface Rights Locations

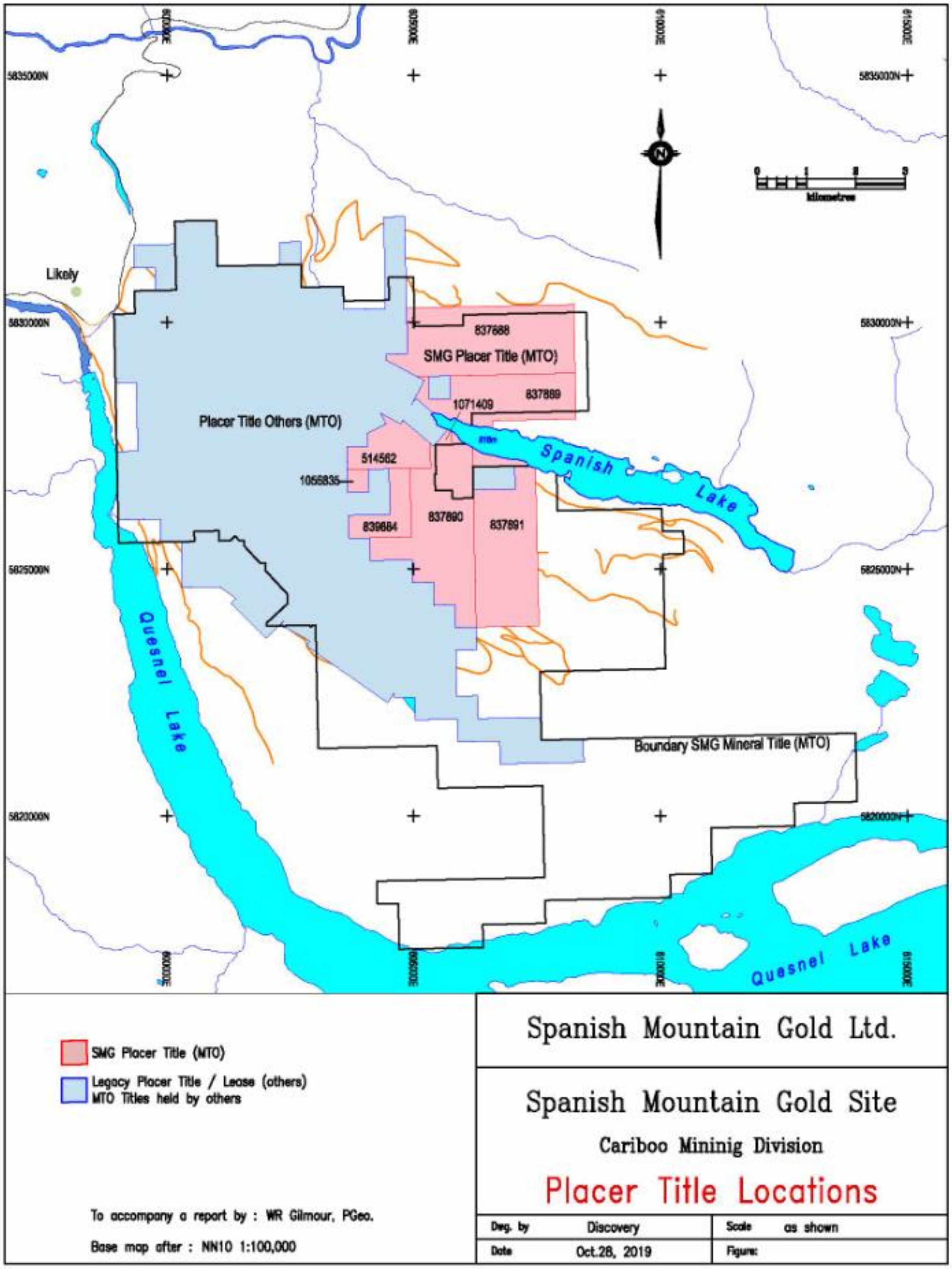


Figure 4-3 Placer Claim Locations

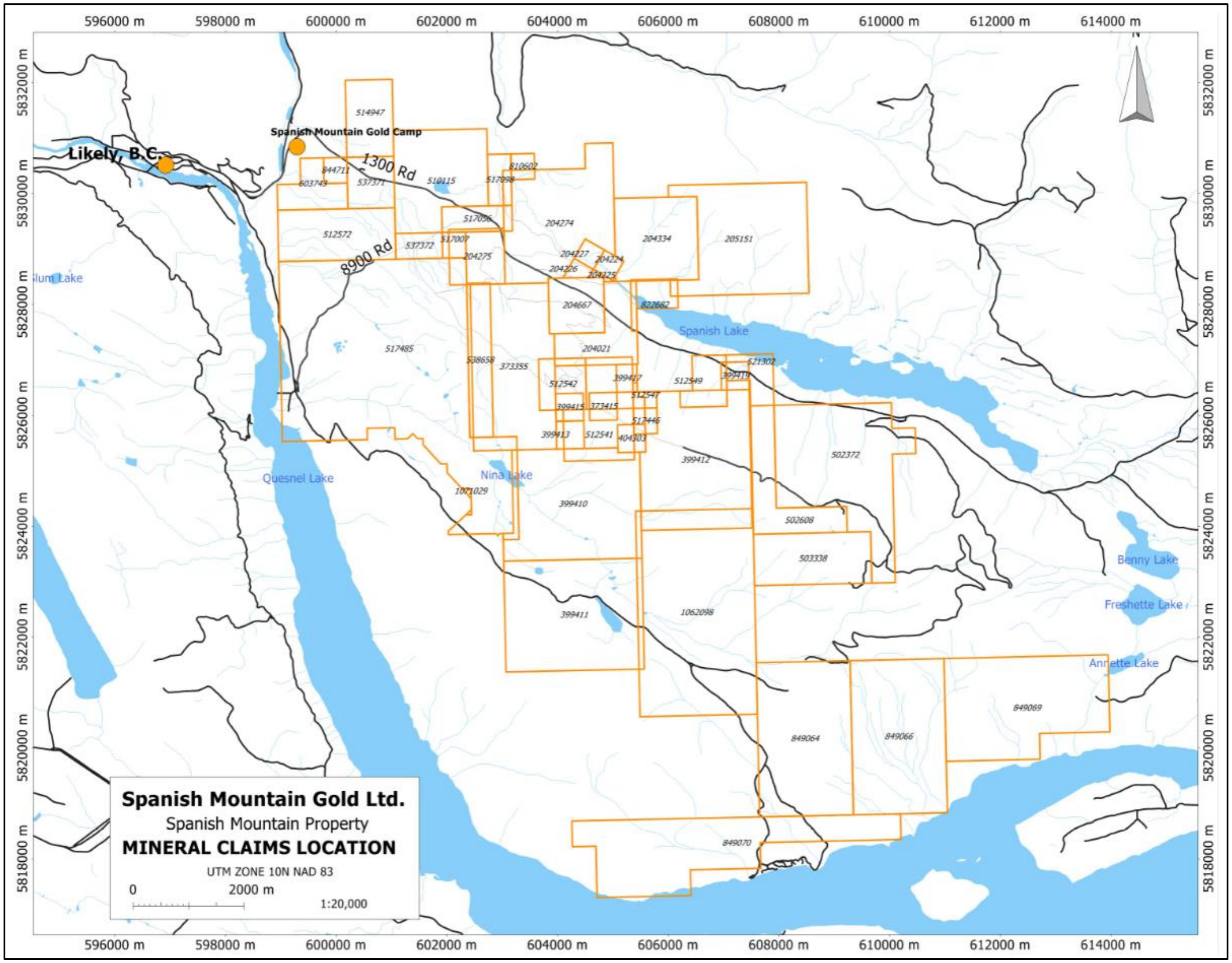


Figure 4-4 **Mineral Title Locations**

5.0 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 a distance of 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. Water is abundant in the area.

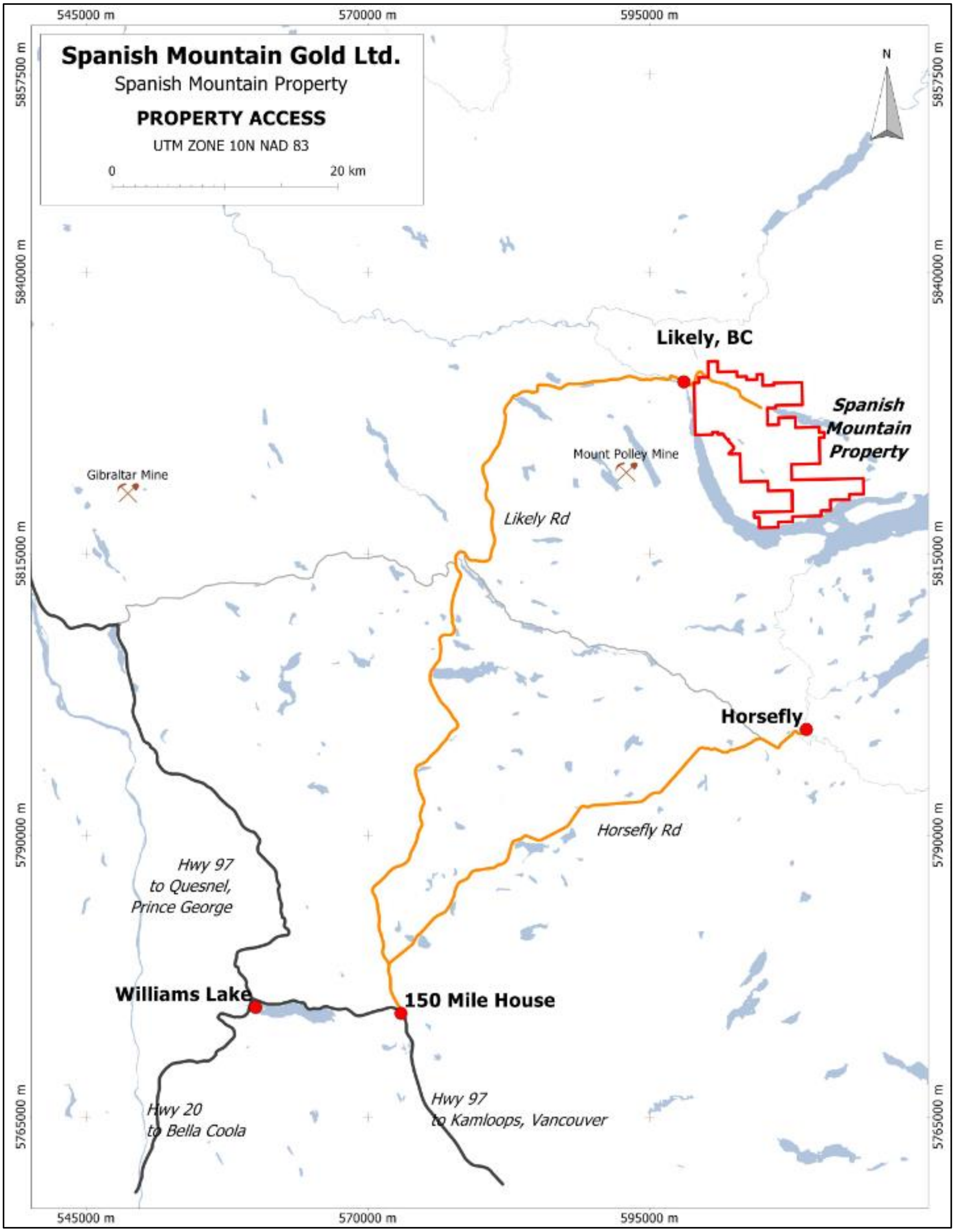


Figure 5-1 Property Access

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 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 disposal, mine waste disposal, 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.

6.0 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.

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)
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

Year	Company	Work Done
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

7.0 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 overlie 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, with the exception of 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.

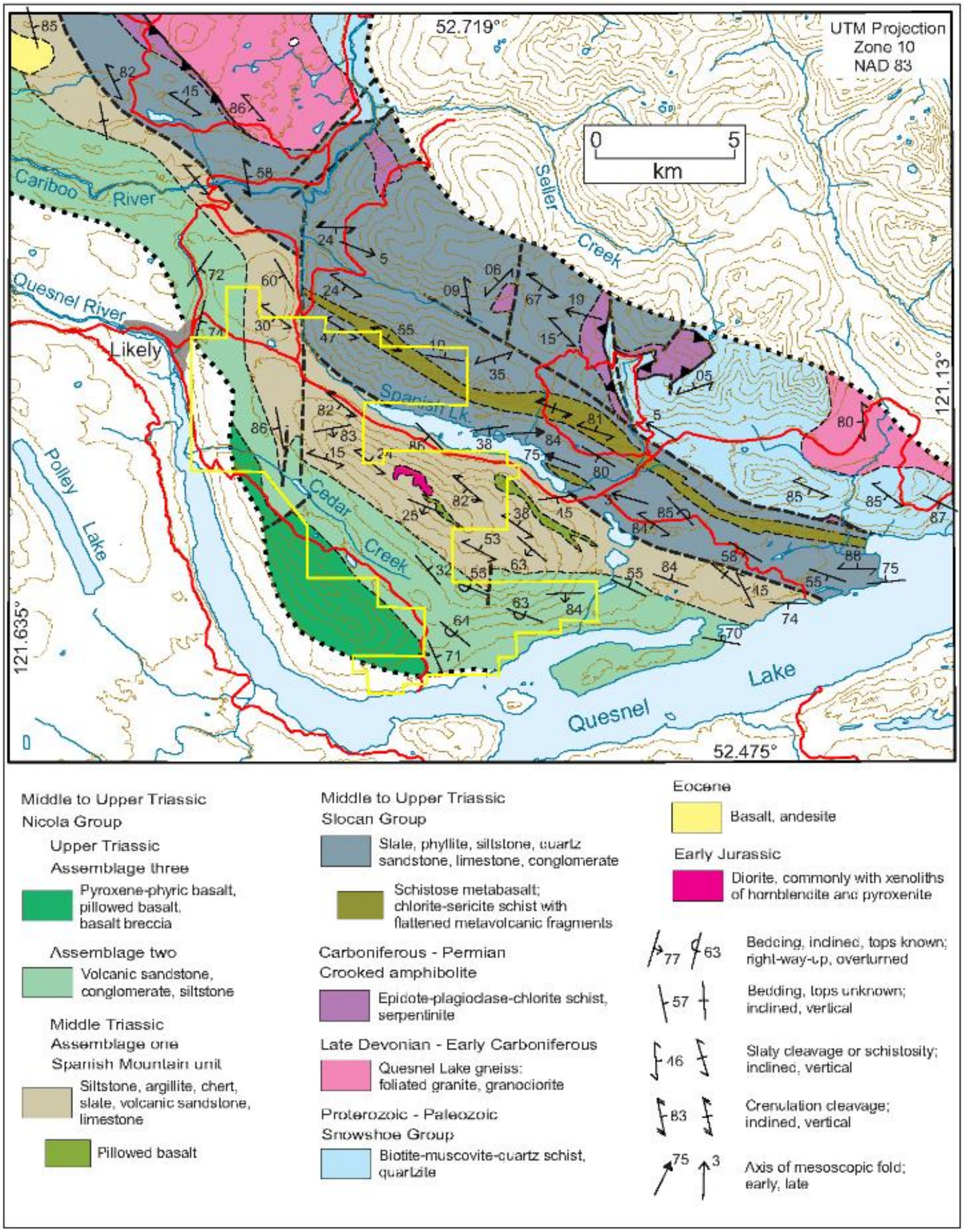


Figure 7-1 Regional Geology, Claims Boundary shown in yellow

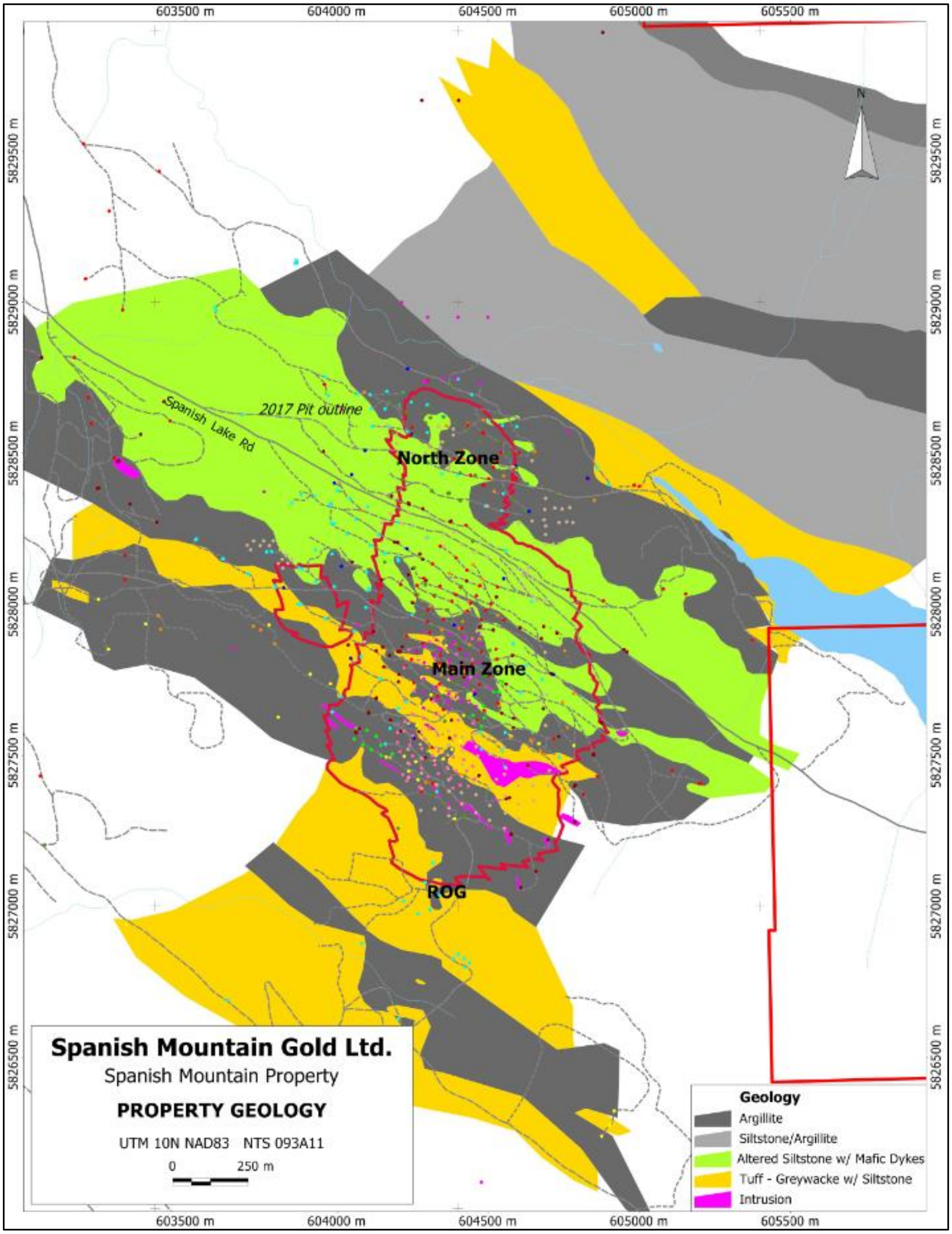


Figure 7-2 Property Geology (approximate pit outline shown in crimson, completely within Claims Boundaries in red)

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 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.

As discussed in Section 7.1, 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, and contains as well 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.

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 of 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 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 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.

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 <20µm, and generally <5 µm. It is associated with equally fine-grained chalcopyrite-galena-sphalerite, which occurs in all the same habits. All of 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 particular 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 and 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.

8.0 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 in particular.
- 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.

9.0 Exploration

This Report is concerned primarily with a Resource for the Main and North Zones, and a Preliminary Economic Assessment, which are based on the results of sampling of both drill core and RC rock chip samples (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. 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 2018 is 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
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. The vast majority 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-element 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. The vast majority of samples were collected on an approximately 50 m by 100 m grid, totalling 36 line km. Values up to 865 ppb gold lead 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 led to 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, from which orthophotos 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, 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.

Soil sampling, totalling 792 samples were 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 is 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 newly exposed, from pad building, outcrops and fault zones in the Main Zone. 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 was 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 in the area of 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 focused 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.

10.0 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 2018.

Table 10-1 Summary of Exploration Drilling Programs on Main and North Zones

Year	Drill Type	No. of Holes	Metres	Core size
2018	RC	11	1,091	n/a
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 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 and 2018 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.

The locations of the 2004 to 2018 drillholes are shown on Figure 10-1 through Figure 10-12. Representative examples of drill sections through the mineral deposit are shown in Figure 10-13 and Figure 10-14.

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 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 [McKeown Placer area] of the LE Pit.

10.2 2005 Program

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 was done 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

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 area 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 drillholes.

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

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 tested 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 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.

No drilling was done in the North Zone; however, re-interpretation of the data suggests 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 done. 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 drillholes; the Cedar Creek area, where two drillholes 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

Definition drilling continued in the Main Zone with a program of 62 core drillholes, totalling 13,769 m. Of these holes, 33 HQ holes were done on 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, totalling 1,705 m. The holes were collared about 200 m apart along a fence oriented from 119° to 289°. The drillholes 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

Drilling comprised 20 core drillholes 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 of 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

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 from the Inferred to 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 drillholes 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 drillholes, 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

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 focused on 131 NQ core drillholes, 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) drillholes 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 drillholes 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

A review by Dr. Morris Beattie, P.Eng. and 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 suggests a negative bias occurred in the sampling from the core drilling.

The 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 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 focused on a test block within the deposit on the Main Zone. In total, 9,226 m were drilled in 56 RC drillholes.

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.

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.

10.11 2014 Program

Additional RC drilling was carried out on the Main and North Zones, 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 measured 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 comprised three metallurgical HQ holes, totalling 512 m, on the Main Zone. The vertical holes were drilled for confirmatory metallurgical testwork. This work is to provide 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 was completed. The core was then stored on pallets and transported by a trucking company to an independent metallurgical lab in Reno, Nevada.

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

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 measured and indicated resources. 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 R8R2K Survey GPS equipment supplied by Cansel Survey Equipment Inc.

10.13 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 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.

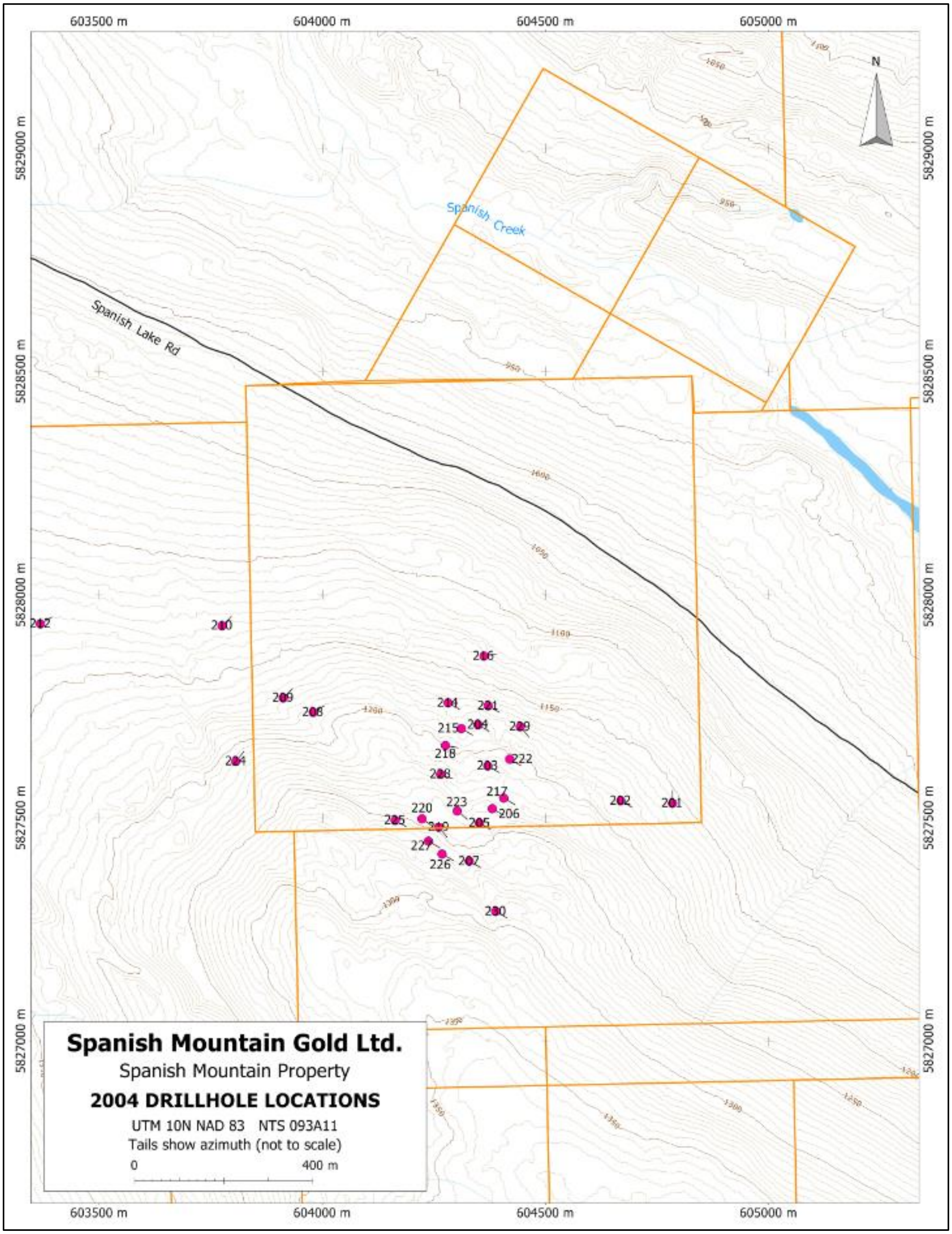


Figure 10-1 2004 Drillhole Locations

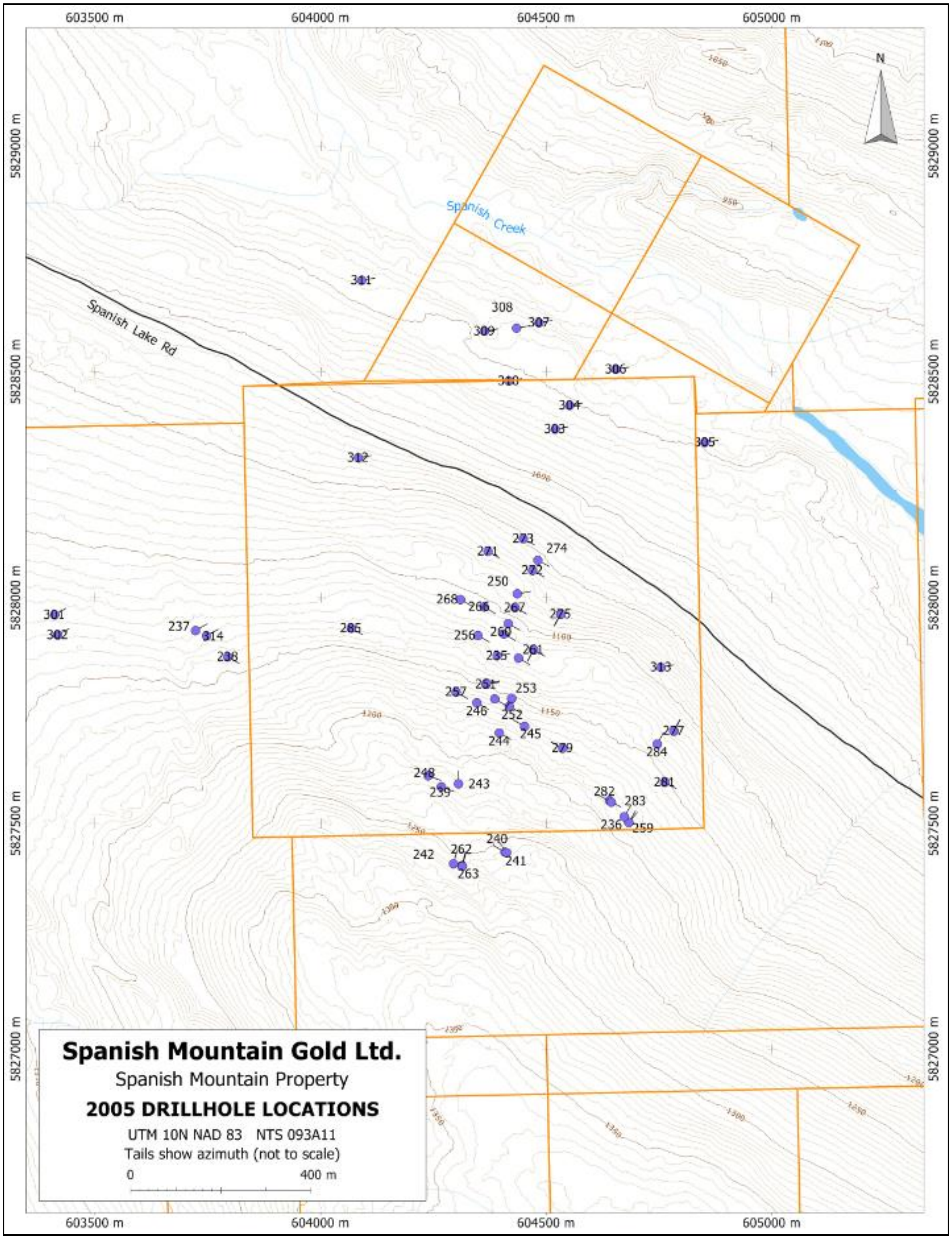


Figure 10-2 2005 Drillhole Locations

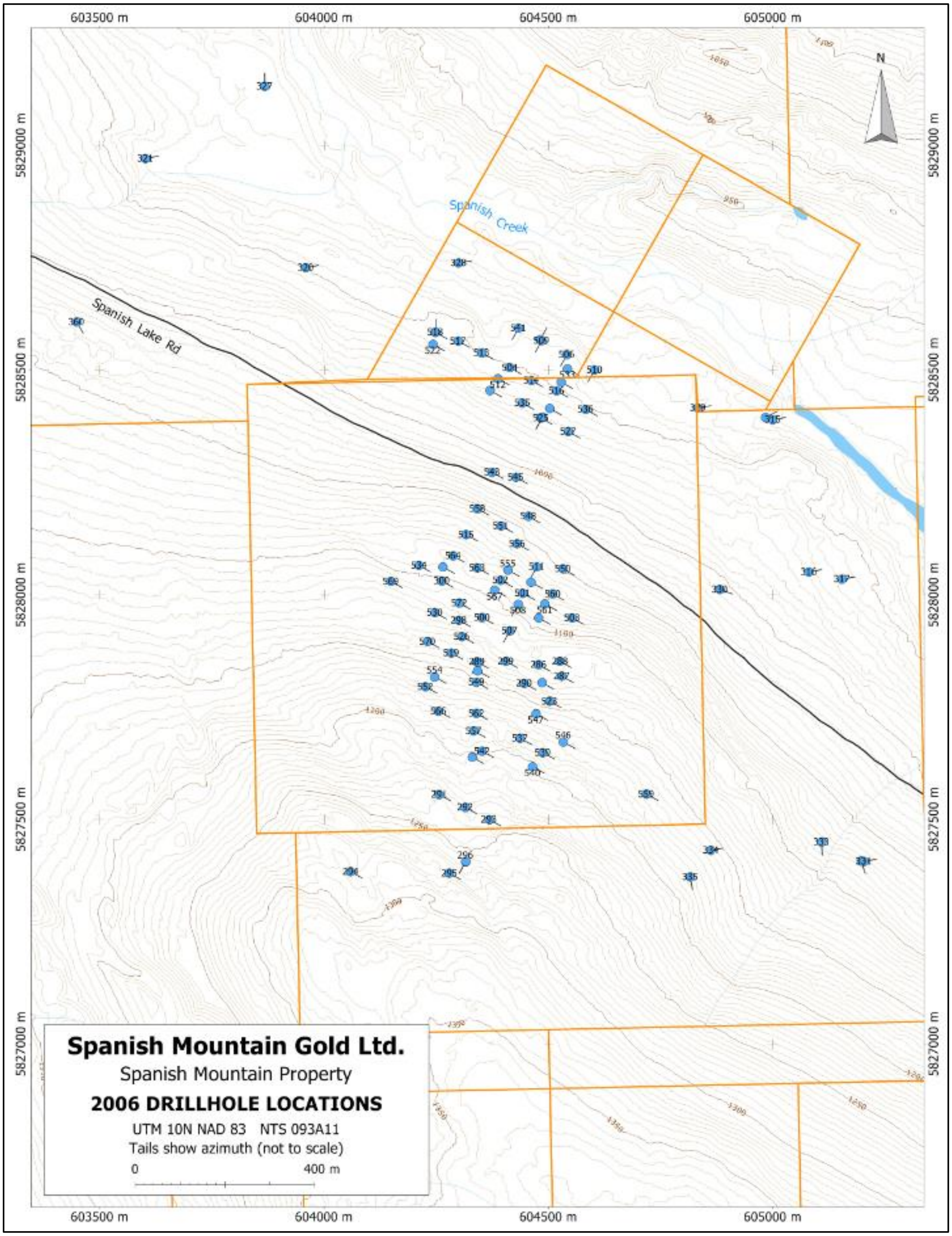


Figure 10-3 2006 Drillhole Locations

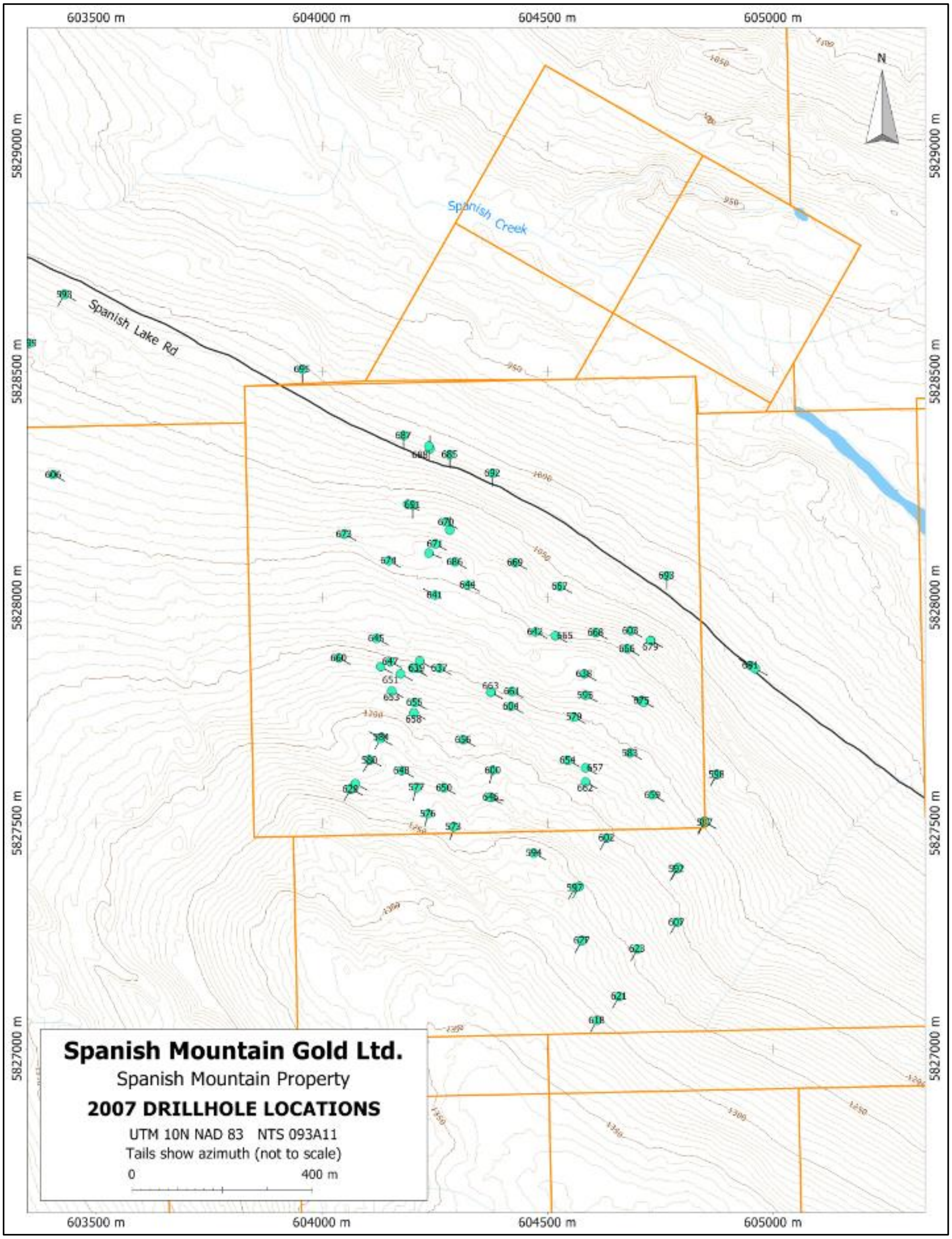


Figure 10-4 2007 Drillhole Locations

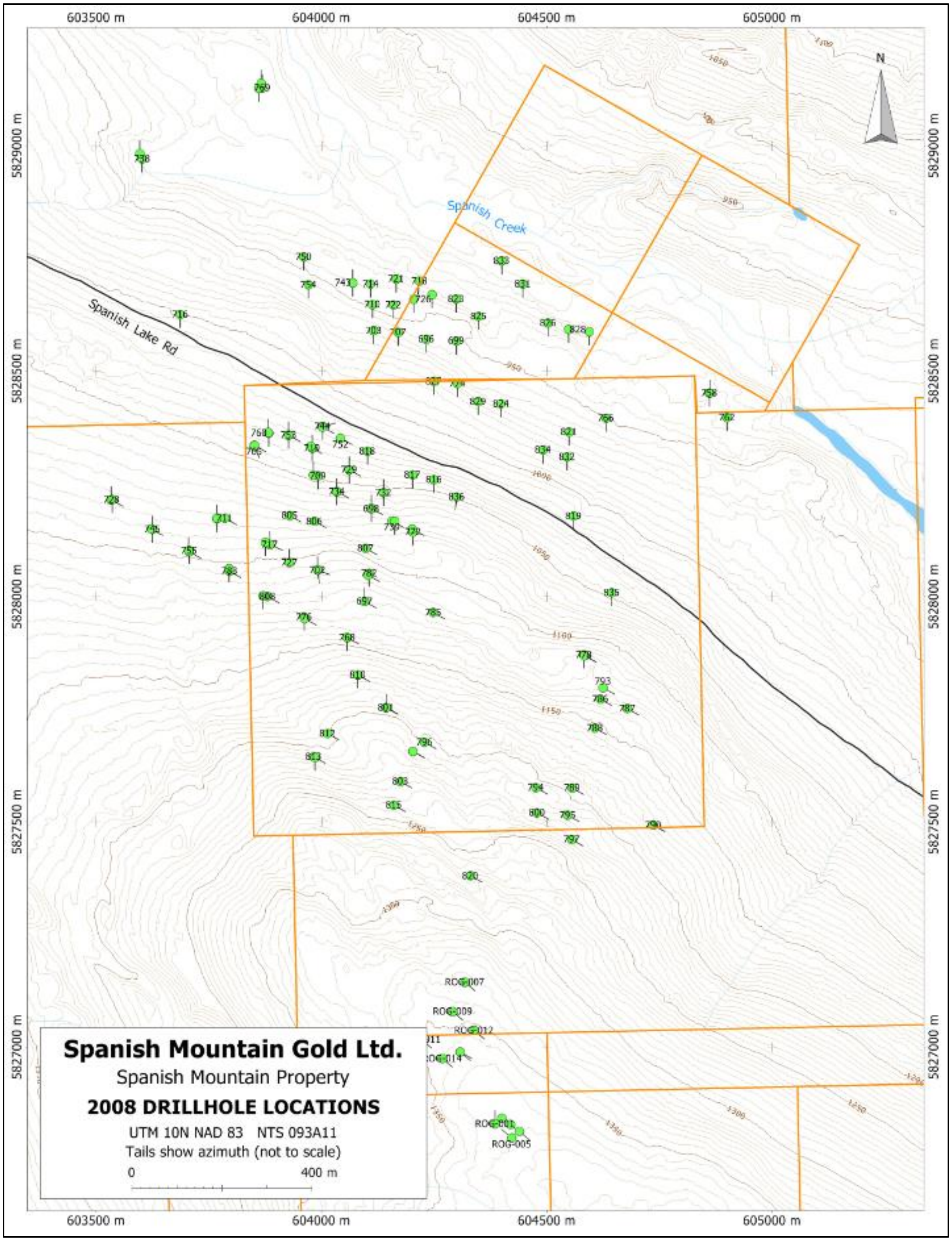


Figure 10-5 2008 Drillhole Locations

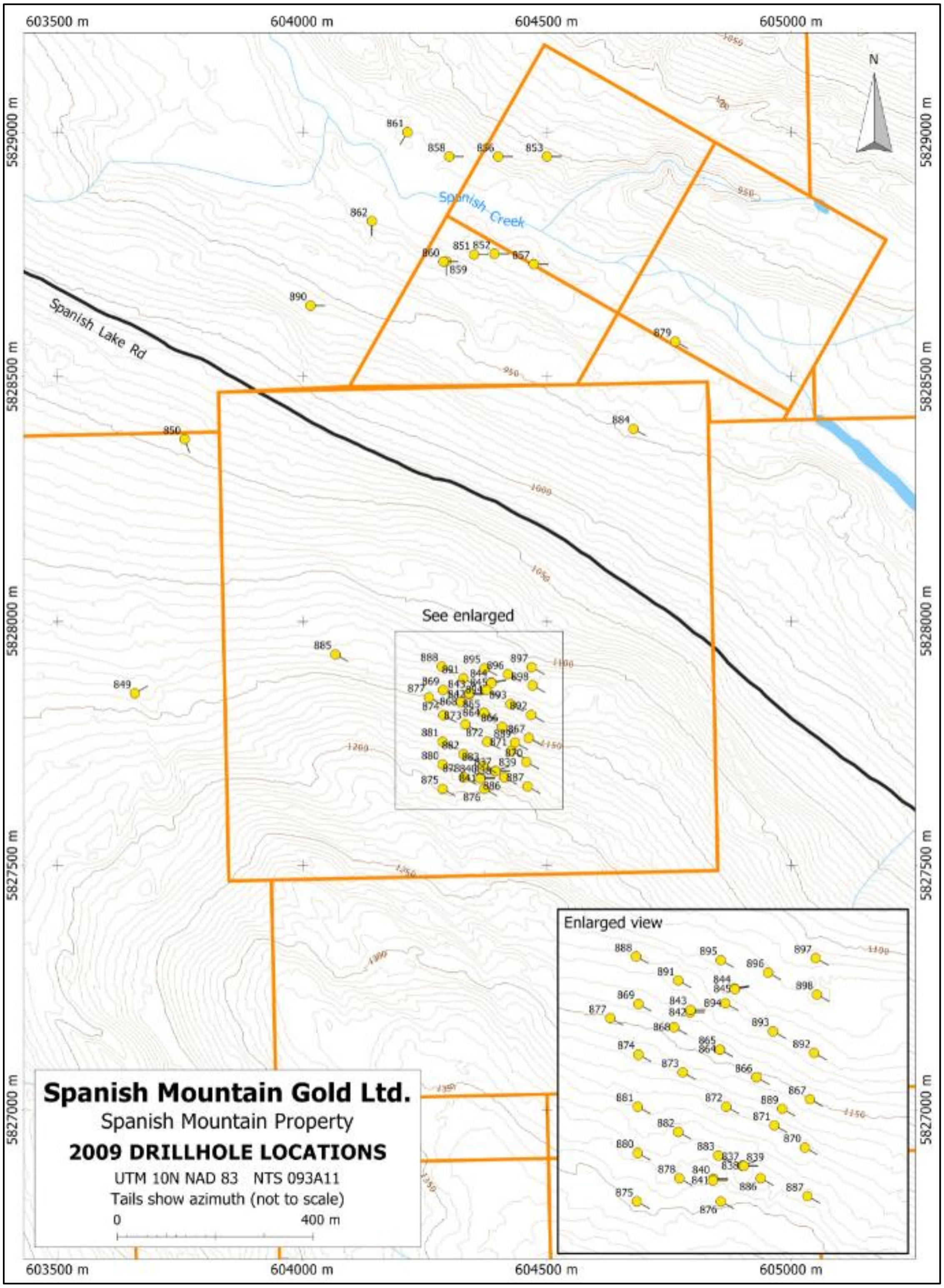


Figure 10-6 **2009 Drillhole Locations**

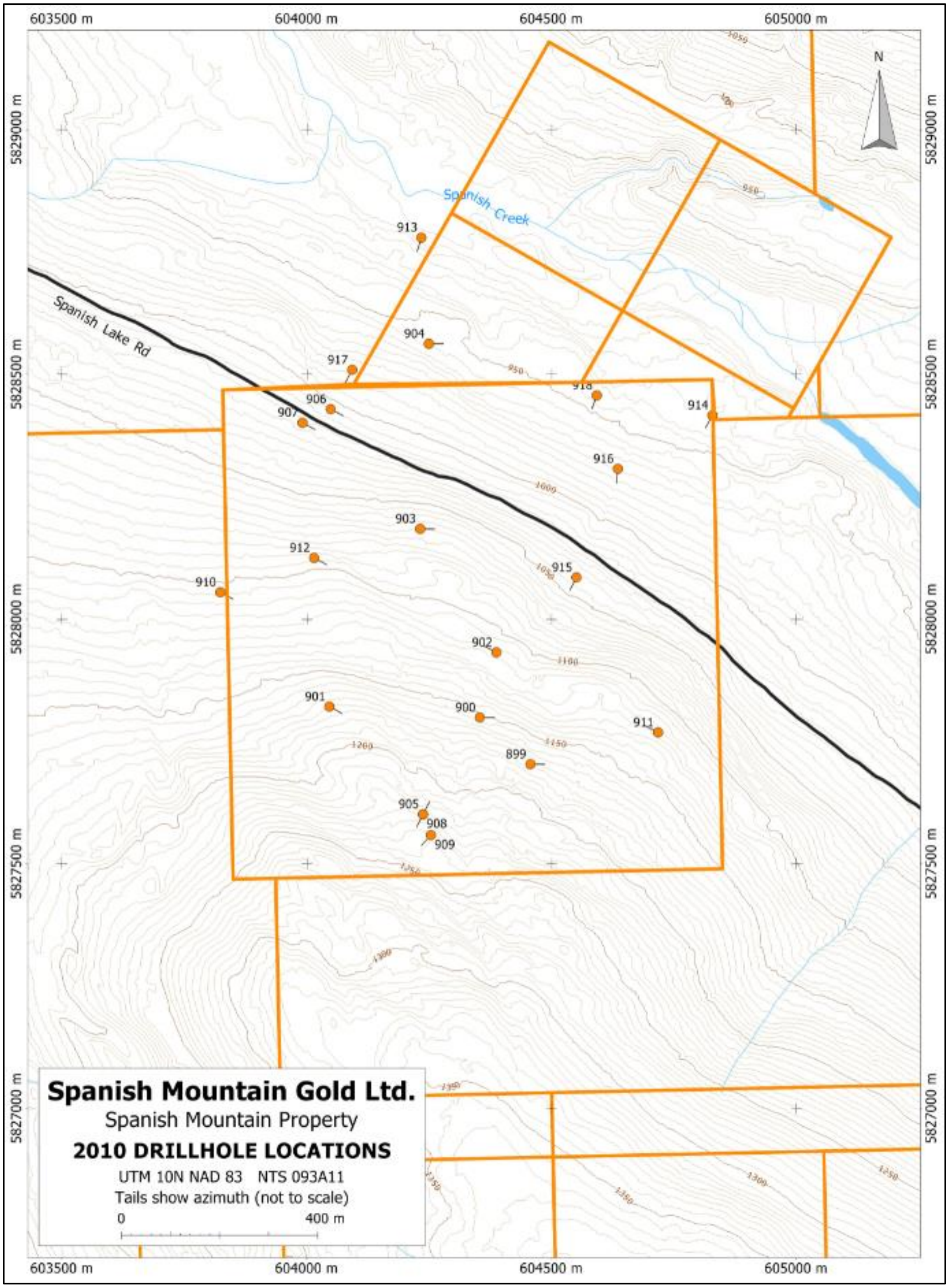


Figure 10-7 2010 Drillhole Locations

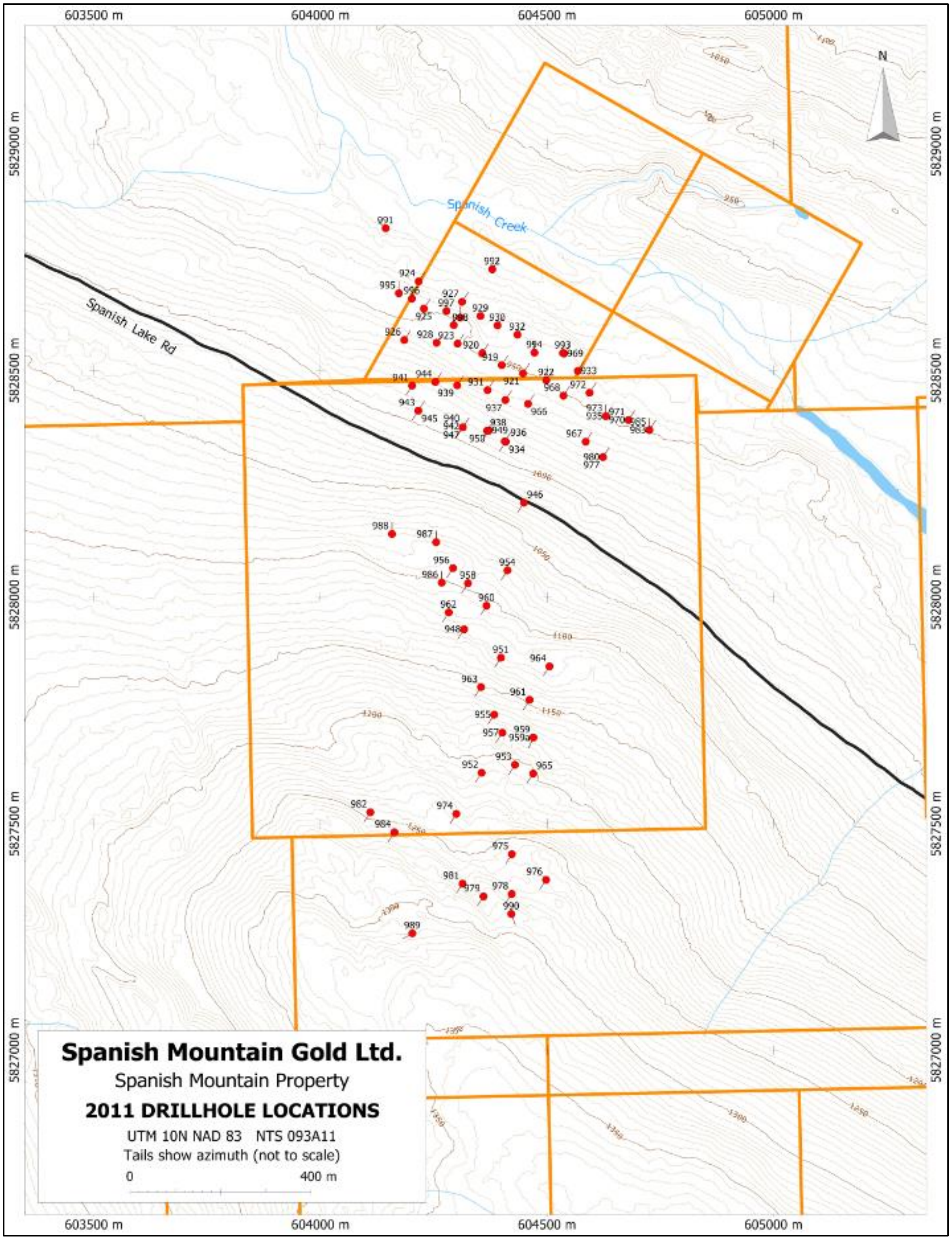


Figure 10-8 2011 Drillhole Locations

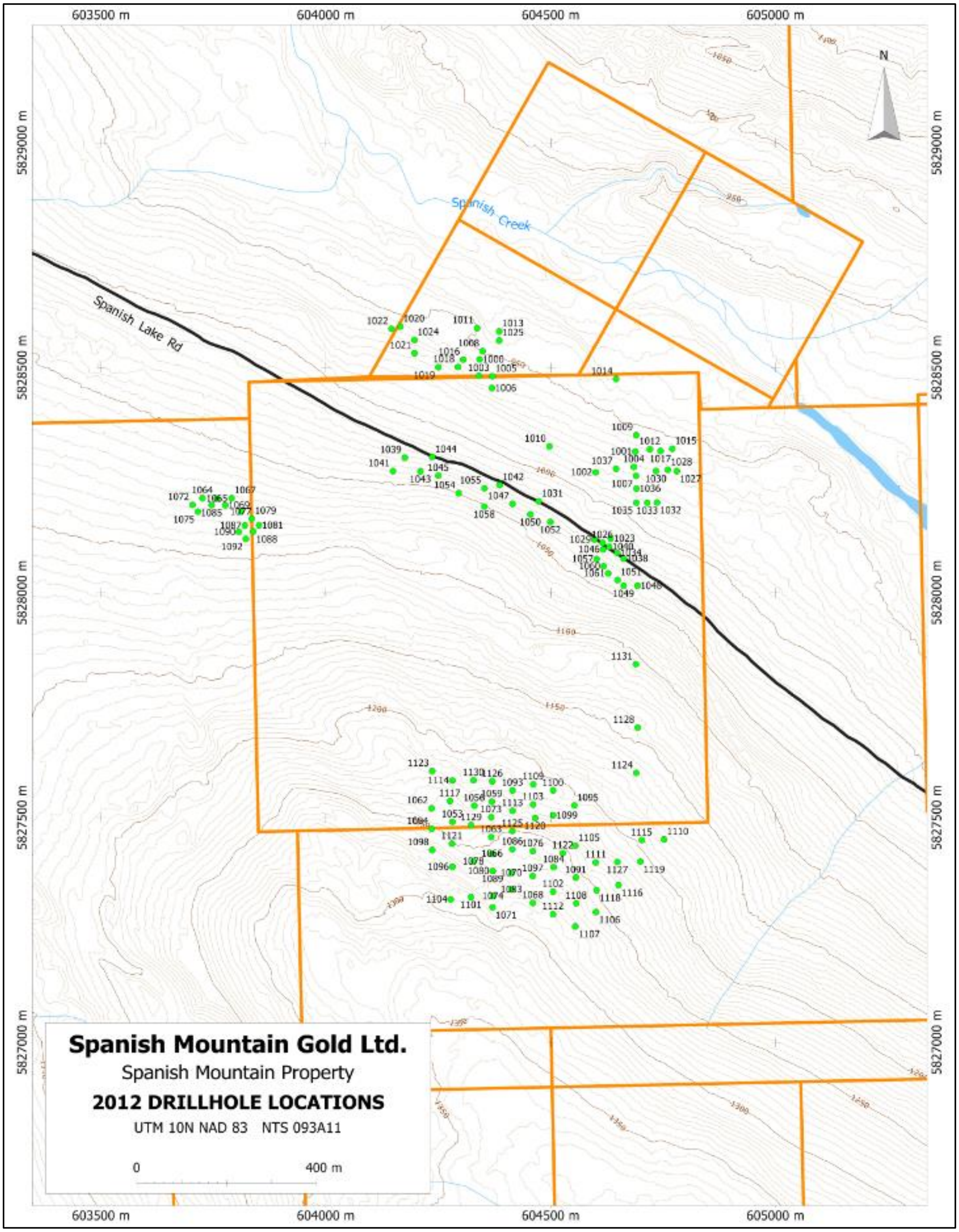


Figure 10-9 2012 Drillhole Locations

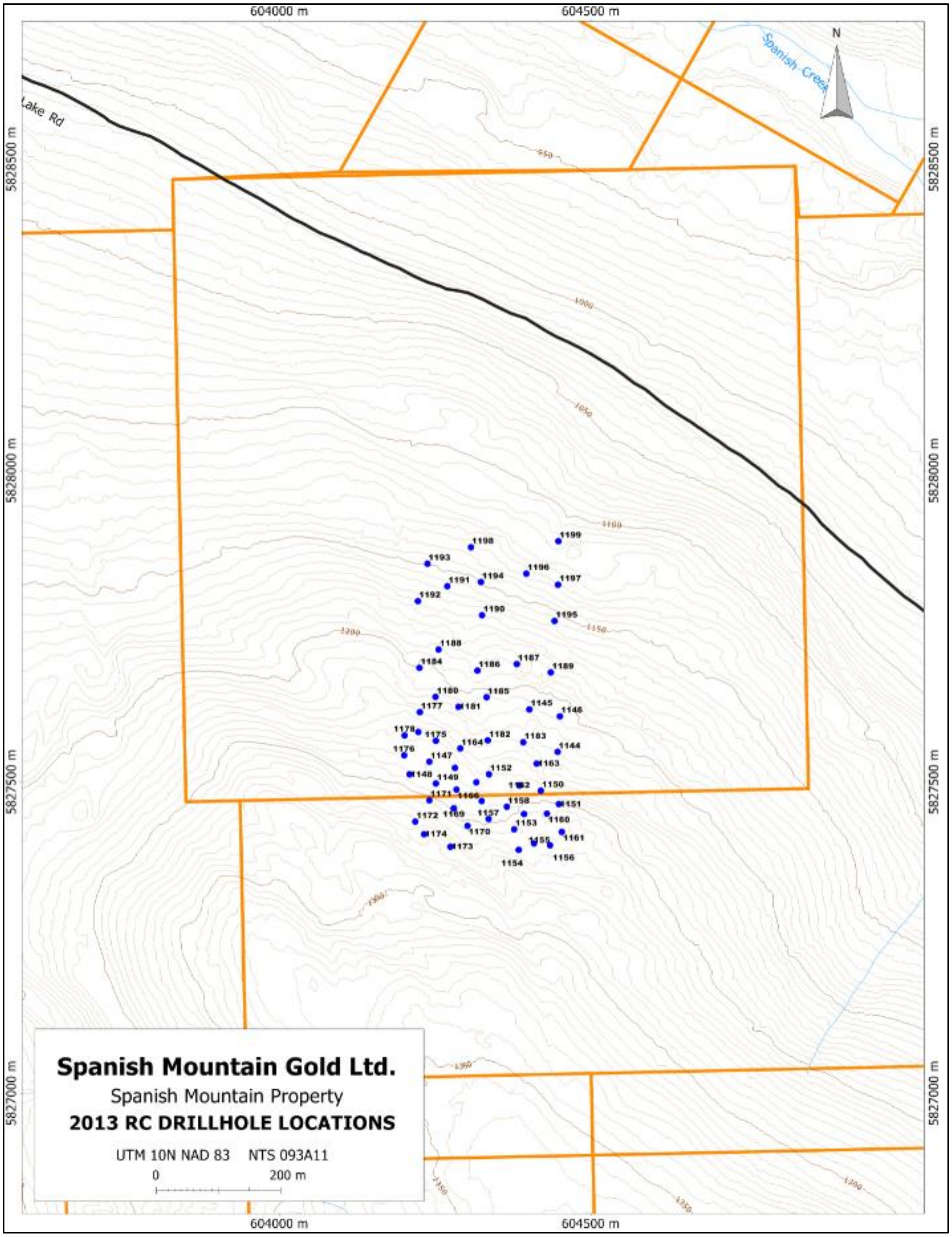


Figure 10-10 2013 Drillhole Locations

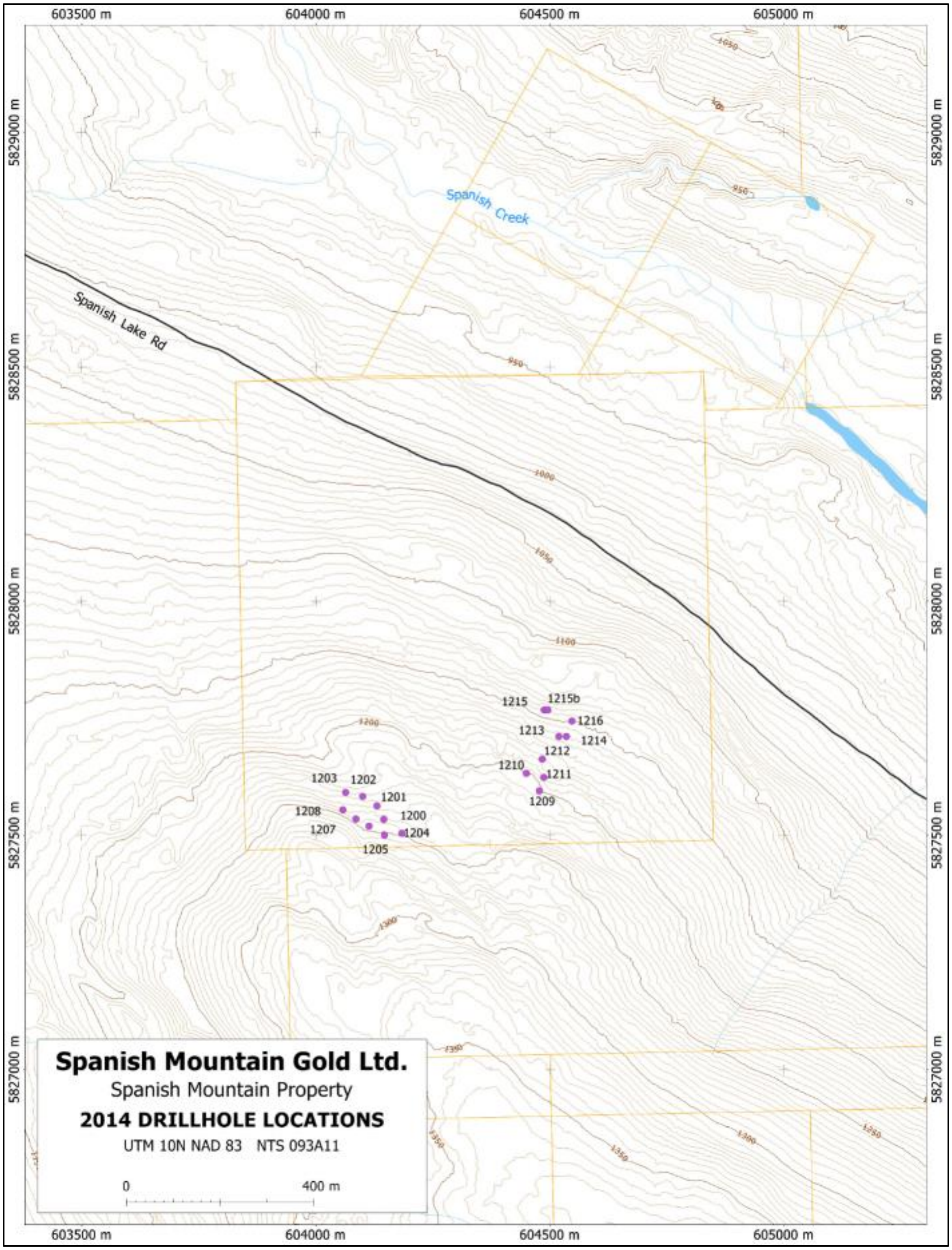


Figure 10-11 2014 Drillhole Locations

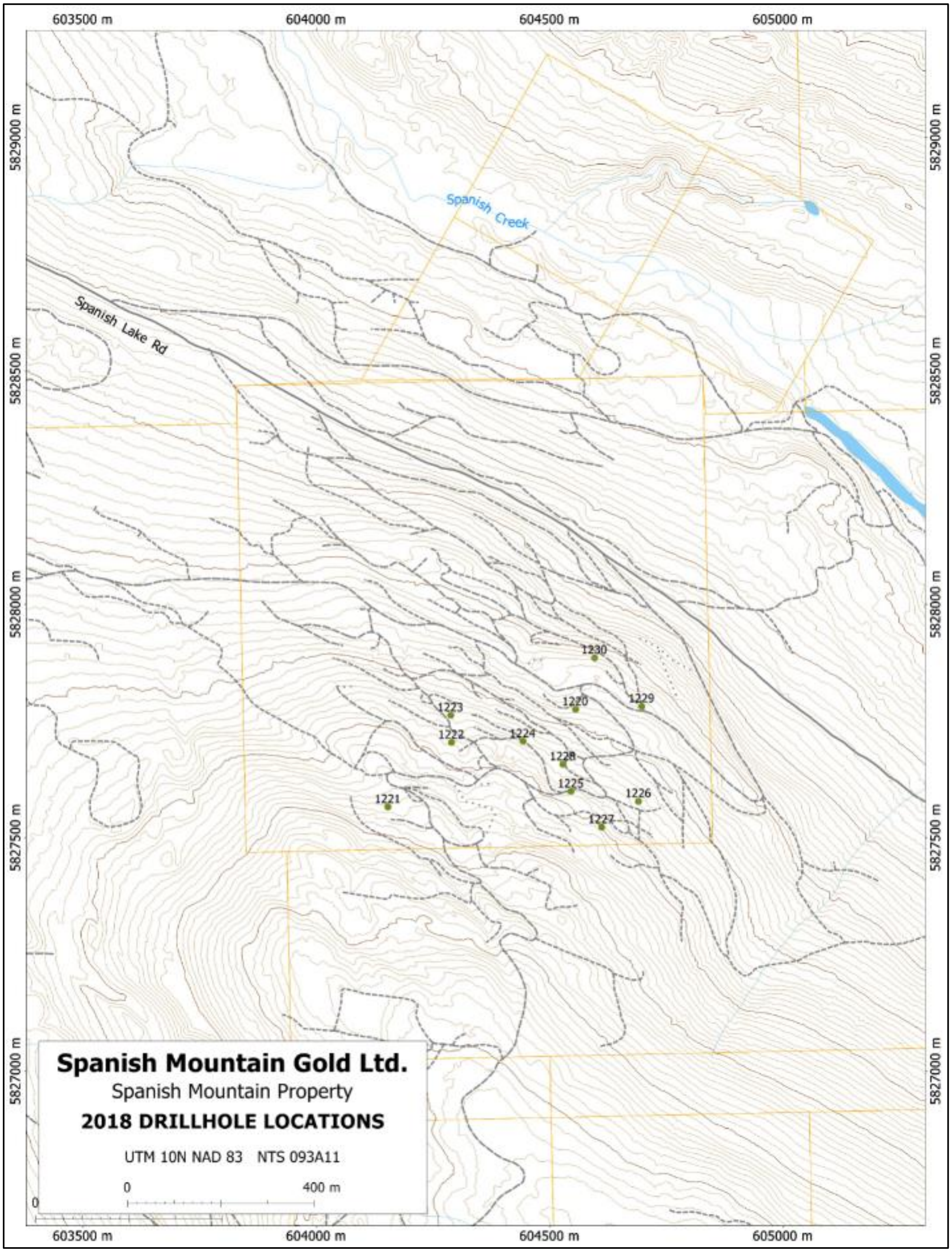


Figure 10-12 2018 Drillhole Locations

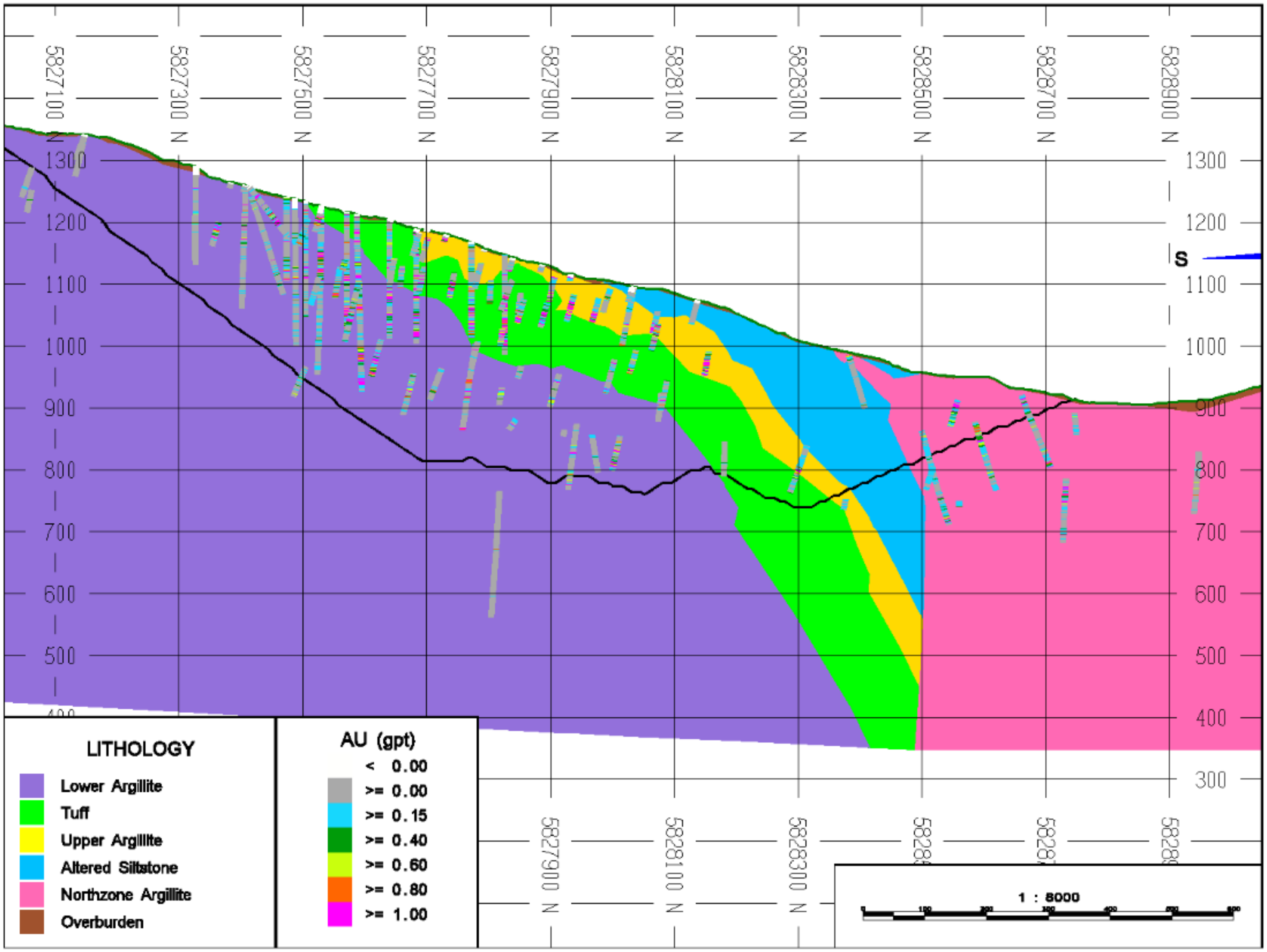


Figure 10-13 Section East 604325 looking West with Drill Grades (+/- 10 m from section) Relative to lithology and Resource Pit Shell

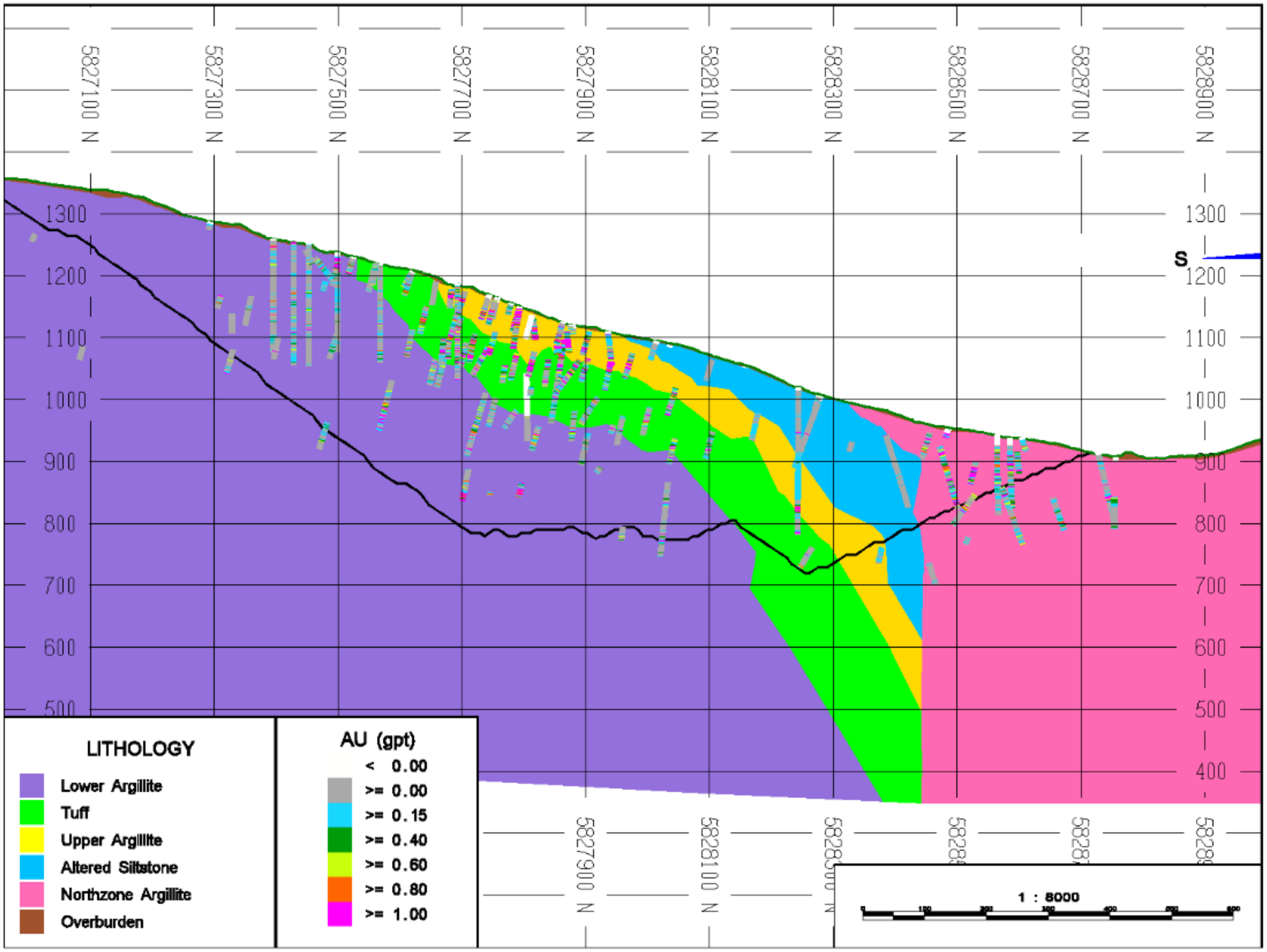


Figure 10-14 **Section East 604385 looking West with Drill Grades (+/- 10 m from section) Relative to lithology and Resource Pit Shell**

11.0 Sample Preparation, Analyses and Security

The following describes the sampling methods used by SMG in the 2010, 2011, 2012 and 2018 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 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.

11.1 Sample Collection and Preparation

11.1.1 2004, 2005 and 2006 RC drilling

Samples were collected every 1.52 m (5 feet) from the cyclone. This 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 lab for analysis and the other retained as a similarly numbered reject, which was 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 lab. The sacks were removed from the field nightly and stored a 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 Labs 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 drill site by pickup truck to 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 often shorter intervals were used in area 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 placed in plastic bags, along with assay tags, which were tied with plastic straps. All samples were stored in a staff facility until shipped to Eco-Tech Labs via either staff, contact personnel, or by Van-Kam Feightways out of Williams Lake.

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

11.1.3 2010, 2011, 2012 and 2018 Core Drilling

The core was transported to SMG's core logging facility, where RQD procedures, core logging, and core sampling and splitting were done. The entire length of the core was sampled. Core was generally sampled in 1.5 m intervals, with shorter lengths given for lithology changes or the presence of visible gold. 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 North Vancouver, BC, for sample preparation and analysis.

The samples and 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 lab duplicates at the lab and 68 core samples. The lab was instructed to process samples in single batches of 80 samples in numerical order to assist with QC/QA protocol.

Drill collar locations were surveyed in-house using Trimble R8R2K Survey GPS equipment.

11.1.4 **2013, 2014 and 2018 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 where it 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-labeled 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 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; however, 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 lab 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 drillhole 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 drillhole 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 lab and 68 rock chip samples. Batches could contain either dry drilled samples, wet drilled samples (now dry) or a combination of both. The lab 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 lab protocol before being mixed to produce a composite sample.

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

Drill collar locations were surveyed in-house using Trimble R8R2K Survey GPS equipment supplied by Cansel Survey Equipment Inc.

11.2 Sample Analysis

11.2.1 2004 and 2005 RC Drilling

For 2004 RC samples, analytical work was performed by Acme Labs, 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 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.

11.2.3 2006 RC Drilling

Eco-Tech carried out the analytical tests. 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.

11.2.4 2007, 2008 and 2009 Core Drilling

Eco-Tech, Amce and ALS of Vancouver, BC, carried out the analytical tests. These labs were ISO certified. Gold analysis and multi-element analysis were completed by the same methods as described in Section 11.2.1.

11.2.5 2010, 2011, 2012 and 2018 Core Drilling

ALS carried out the analytical tests. Gold analysis and multi-element analysis were completed by the same methods as described in Section 11.2.1.

11.2.6 2013, 2014 and 2018 RC Drilling

ALS carried out the analytical tests. Gold analysis and multi-element analysis were completed by the same methods as described in Section 11.2.1.

11.3 Sample Security

Drill core/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

Two difference 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 in available.

11.4.2 2005 Core 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, 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

QC/QA comprised the insertion of analytical standards, sample blanks and preparation sample duplicates, at a rate of one each 35 samples, into the sample stream. 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 average and 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 lab from a cut of the crushed sampled. The specific procedures of the lab 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

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, 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.

11.4.6 2010, 2011 and 2012 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 in ALS's lab 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 from the lab 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**

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 of the sample stream, gave results within acceptable tolerances, demonstrating 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 due to the significantly larger sample collected by the RC drilling, with both 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 consisted of 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 lab. 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 lab 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 from the lab include 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.

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 2004 and 2005 RC drilling, as a precaution against contamination, the splitter and buckets were cleaned out between each sample, and the cyclone 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 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 and 2012 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 – 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 lab, so that each sample follows 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 not as contamination from mineralized drill cuttings. They were also generally not near mineralized zones, hence there was no effect on resource sections. 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 in order 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, in order 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 on sections of several drillholes; at Acme and ALS. Minor variations occurred locally, but the overall Au grade did not change significantly. These data are not available to the authors of this Report.

In 2004, no duplicate field or preparation samples were produced. A few duplicate pulp samples were analysed by the lab. This report does not compile those results.

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

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; 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.

The following Table 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 example, the other half of the core) 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 the following Table.

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 the following Table.

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), and another 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 the following Table.

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.

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 selected samples, one in every 40 (173 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 cuttings. 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 the following Table.

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

For the 2010, 2011 and 2012 drilling, ALS prepared two 30 g sub-samples per sample for every sample of core, producing 15,317 pairs. Sample pairs containing an average grade of at least 0.04 g/t Au (7,278 pairs) were plotted by the Thompson-Howarth method. The results are summarized in the following Table.

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 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 per sample 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 the following Table.

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 in the field. 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 that 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

Two difference 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 in available.

No conclusions can be drawn as to accuracy for the 2004 drilling.

11.7.2 2005 Drilling

Two different standards were inserted into the sample stream in the field. 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

Two different standards were inserted into the sample stream in the field. Only one sample was outside of the acceptable limits. No abnormal trends or material bias were noted.

11.7.4 2007 Drilling

Three different standards were inserted into the sample stream in the field. 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

Three different standards were inserted into the sample stream in the field. Only three samples were outside of the acceptable limits. No abnormal trends or material bias were noted.

11.7.6 2009 Drilling

Six different standards were inserted into the sample stream in the field. 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 2018 Drilling

All but one of the SMG inserted gold standards 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 three 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 inserted into the sample stream at the rate of one every 20 samples; that is, 4 standard samples in each 80-sample batch. 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 inserted into the sample stream at the rate of one every 20 samples; that is, 4 standard samples in each 80-sample batch. 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 of the results included plotting the results for each SMG and ALS standard in order of report completion. The charts were regularity 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; however, no changes in the 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.

12.0 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 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, QC/QA procedures; reviewing analytical certificates throughout the drill program. The co-author William Gilmour, P.Geo., was responsible for reviewing the results, including QC/QA, 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.

For this Report, a selection of data from signed laboratory analytical certificates for the 2004 to 2018 drilling was compared with the digital data used in the resource calculation. No significant discrepancies were noted.

12.1 Comments on Section 12

The process of reviewing the data used in the mineral resource estimate have been reviewed by QP William Gilmour, P.Geo., and in his opinion sufficient verification checks have been undertaken on the databases to provide confidence that the databases are reasonably error free and may be used to support the Mineral Resource estimation.

13.0 Mineral Processing and Metallurgical Testing

Several metallurgical test work phases have been completed between 2007 and 2019 at the following independent laboratories:

- G&T Metallurgical Services (G&T) in Kamloops BC
- SGS Minerals Services (SGS) in Lakefield ON
- Knelson Research and Technology Centre (Knelson Research), Langley BC
- Met-Solve Laboratories, Langley BC
- McClelland Laboratories Inc., Sparks NV

The test work programs and reports listed in Table 13-1 are a chronology of all the test programs conducted on the SMG resource to date.

Table 13-1 Test Work Programs and Reports

Document or Test Program	Author or Laboratory	Date
Petrographic Study of the Spanish Mountain Project, Cariboo Mining District, British Columbia	Panterra Geoservices Inc	October 5, 2006
Preliminary Metallurgical Assessment of Samples from the Spanish Mountain Project, Report No. KM1921	G&T	November 28, 2007
Cyanidation Test on Flotation Concentrate, Report No. KM2138	G&T	December 12, 2007
Mineral Processing Review of Spanish Mountain Project for Skygold Ventures Ltd	Westcoast Mineral Testing Inc	January 4, 2008
Progress Report No. 1, Spanish Mountain Gold Project, Report No. KM2637	G&T	August 30, 2010
Comparative Gold Content in Core Using Gravity Concentration Techniques – Spanish Mountain Project, Report No. KM2538	G&T	April 2010
Metallurgical Test Report – Spanish Mountain Gold, KRTS 20559	Knelson Research	May 19, 2010
Gravity Concentration and Flotation of Spanish Mountain Composites, Spanish Mountain Gold	M. Beattie	September 2010
NI 43-101 Technical Report- Preliminary Economic Assessment for the Spanish Mountain Project	AGP	December 20, 2010
Grinding Circuit Design for the Spanish Mountain Project Based on Small-Scale Data, Project 12488-001 – Report 1	SGS	December 23, 2010
Spanish Mountain Gold Project Process Development- Summary Report to September 2011- Client Memorandum to Tetra Tech	M. Beattie	September 2011
Memorandum Updates	M. Beattie	Various dates October 2011 to July 2012
Metallurgical Testing on Samples from the Spanish	G&T	September 7, 2011

Document or Test Program	Author or Laboratory	Date
Mountain Gold Project, Report No. KM2637		
Gravity Modelling Report – Spanish Mountain Gold, KRTC 20559-1	Knelson Research	October 18, 2011
A Variability Test Program on Samples from the Spanish Mountain Deposit, Project 12488-002- Report #2	SGS	March 19, 2012
Metallurgical Testing on Variability Samples from the Spanish Mountain Gold Project, Report No. KM3185	G&T	June 21, 2012
MS1735 Metallurgical Testing without gravity and without carbon prefloat.	Met-Solve	March 26, 2017
Metallurgical Testing - Spanish Mountain Drill Core Composites MLI Job No. 4373	McClelland Laboratories Inc.	November 20, 2019

The metallurgical test work has used to characterize the ore, develop a flowsheet, test the metallurgical performance, and develop process parameters.

Test work was carried out on samples collected from the SMG exploration drillholes and included the following:

- comminution characterization
- evaluation of whole-ore cyanide leaching
- gravity concentration
- flotation optimization
- gravity concentration with flotation of gravity tailings
- carbon rejection with pre-flotation and cleaner flotation using CMC
- cyanide leaching of gravity concentrates
- cyanide leaching of flotation concentrates
- cyanide leaching of gravity middlings and recombined middlings/tailings
- gravity scavenging of cleaner and recleaner tailings

13.1 Mineralogy

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. The deposit contains carbonaceous material, graphite, which requires rejection prior to leaching to prevent preg-robbing during leaching.

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.

G&T report KM1921 showed a scarcity of minerals containing copper, lead, zinc, arsenic, antimony and other trace elements. An average of 22% of the carbon present occurring was in organic form.

13.2 Sample Head Grade

Head grades of various test samples have been characterized by assay and the results have been detailed in several reports.

G&T report KM1921 presented the results of thirteen composite samples compiled from various drill cores. These samples were then combined to create three master composite samples and one master composite blend sample. Head assays of these samples ranged from 0.82 g/t gold to 7.48 g/t gold. Feed compositions of the four master composite samples are shown in Table 13-2.

Table 13-2 Head Composition – G&T KM1921

Composite ID	Assays								
	Cu (%)	Fe (%)	Mo (%)	As (ppm)	Ag (g/t)	Au (g/t)	S (%)	C (%)	TOC (%)
Master 1	0.01	4.37	<0.001	<10	1	0.62	0.98	2.75	0.19
Master 2	0.01	3.72	<0.001	<10	5	1.18	2.33	2.89	0.87
Master 3	0.01	3.86	0.001	25	4	2.00	2.00	2.53	0.77
Master 1, 2, 3 Blend	-	4.49	-	-	1	1.18	2.04	2.71	-

G&T KM2538 reports on a test program designed to determine the gold content of 148 core intervals using mineral processing to minimize the effect of nugget-bearing gold samples using gravity concentration. Metallurgical assays had head grades ranging from 0.02 to 6.20 g/t gold.

G&T progress report KM2637 had three master composite samples created from one drillhole located in the starter pit area of the deposit. Gold and TOC grades varied in the samples which allowed for variation in the samples for testing purposes. The gold grades were lower than for the previous master composites and were more representative of anticipated mill feed. Table 13-3 shows the assay values obtained for the sample material tested.

Table 13-3 Feed Composition – G&T KM2637

Composite ID	Assays							
	Au (g/t)	Ag (g/t)	Fe (%)	S_{total} (%)	S^{2-} (%)	S as SO_4 (%)	TOC (%)	C (%)
865-1 Rhyolite Tuff	0.45	1.2	4.81	1.40	1.30	0.02	0.28	3.31
865-2 Argillite	0.94	1.2	4.12	2.96	2.88	0.03	1.18	3.22
865-3 Rhyolite Tuff	0.82	0.9	3.32	1.49	1.39	0.02	0.26	2.31

Variability testing was conducted at two laboratories: G&T and SGS. SGS gold values ranged between 0.24 and 1.88 g/t gold and averaged 0.60 g/t gold; TOC values ranged between 0.48 and 1.57% TOC and averaged 1.69% TOC.

The G&T equivalent values varied more widely; values ranged between 0.03 and 1.68 g/t gold and averaged 0.45 g/t gold, while TOC values ranged between 0.03 and 2.03% TOC and averaged 0.89% TOC.

Additional metallurgical testwork was carried out in 2019 utilizing drill core interval samples, from drillholes 18-DH-1217, 1218 and 1219. The samples included argillite, siltstone and greywacke rock types. Select intervals were used for comminution testing. Intervals from drillholes 18-DH-1217 and 1218 were each prepared and assayed. Based on assay results, a metallurgical composite was prepared from each of the two drillholes. All available drill core from hole 18-DH-1219 was composited to prepare a third, low grade metallurgical composite.

Head analyses showed that the 18-DH-1219 composite (Composite 4373-001) was very low in grade (0.36 g/t Au). The current mine plan minimum annual average Au grade is 0.66 g/t and Composite 4373-01 would therefore be considered waste rock. Average head grades for the 18-DH-1217 composite (Composite 4373-002) and the 18-DH-1218 composite (Composite 4373-003) were 0.82 and 1.02 g/t Au. Composite 4373-004 was an equally weighted master composite for bulk testing, prepared from Composites 4373-002 and 003. Predicted head grade for that composite, based on interval assays, was 0.95 g/t Au. Composites 4373-002, 003 and 004 ranged in sulphide sulfur grade from 1.61% to 1.70% and in organic carbon content from 0.72% to 0.95%.

Samples provided for the various metallurgical test programs are generally representative of potential SMG mill feed.

13.3 Grindability

G&T prepared three composite samples from drillhole DDH 865 in early 2010. Bond ball mill work index (BWi) tests were conducted on each of these composites and the values are presented in the progress report KM2637.

An additional 24 variability samples were taken from drillholes across the deposit and processed at SGS in 2010. The variability samples classified by rock domains have been tested in the following grindability tests:

- Bond low-energy impact test (Bond crushing work index (CWi))
- SAG Mill Comminution (SMC) test
- Bond rod mill index (RWi) grindability test performed at a grind of 1,180 µm
- BWi grindability test performed at a grind of 212 µm
- Bond abrasion index (Ai) test.

Grindability results have been summarized in Table 13-4.

Table 13-4 Summary of Grindability Results by Rock Type – SGS and G&T

Rock Type	Average RWi	Average BWi	Average Ai	Average CWi
Argillite	13.4	12.8	0.229	10.9
Tuff	14.7	12.7	0.199	13.9
Siltstone	15.3	15.4	0.269	12.6
Crystal Tuff	16.7	15.6	0.244	15.4

The deposit consists approximately of 50% argillite and 50% non-argillite rock types which consist mainly of tuff rock type. The siltstone component will not exceed 5% of mill feed based on the current mine plan and mineralized crystal tuff samples have all been below the expected cut-off grade.

Grindability results indicate that mill feed for Argillite samples are moderate to soft material hardness. Ai values obtained range from 0.111 to 0.299 g, which classifies the abrasiveness of the samples as mild to medium.

In addition to the work index determinations, SGS carried out JKTech drop weight tests, or SMC tests, on each composite. This series of tests confirmed that the softest of the rock types is the argillite. The JKTech drop weight tests also indicated that a pebble crusher would be required in closed circuit with the semi-autogenous grinding (SAG) mill if a (SAG) mill is used for grinding.

In 2019 McClelland's carried out crusher work index, abrasion index and ball mill work index (Bond method) tests on seven samples from the three drillholes. A ball mill work index test was also conducted on each of Composites 4373-001, 002 and 003 with 100-mesh (150µm) closing screens. Ball mill work indices ranged from 11.5 to 12.6 kWh/t with an average of 12.2 kWh/t, which can be characterized as moderate hardness. Crusher work index ranged from 4.9 to 12.2 kWh/t with an average of 9.5 kWh/t, which can be characterized as soft. Average abrasion index was 0.17 g, which can be classified as moderately abrasive.

13.4 Whole Ore Gravity Concentration

Various gravity concentration test work programs have been conducted during the metallurgical test programs.

Gravity recovered concentrate was generally found to have a lower amount of carbon associated with it, and, as such, the gold recovery via leaching has been relatively high, with up to 98.6% gold recovery realized from leaching gravity concentrates after regrinding.

Gravity concentration results from G&T Reports KM2538 and KM2637 have been extensively analyzed by SMG. The average recovery of gold to the gravity concentrate in this test work was 42% for the non-argillite samples, and 26.3% for the argillites, or 34.1% as an overall average.

In 2010, Knelson Research was provided with two composite samples for Extended Gravity Recoverable Gold (EGRG) testing. The EGRG test procedure consists of sequential grinding and recovery stages to establish the amenability of the material to gravity concentration. Two different samples from the center of the Main Zone were provided for this program. EGRG recovery results are summarized in Figure 13-1 including a corrected 865-3 to account for a gold nugget.

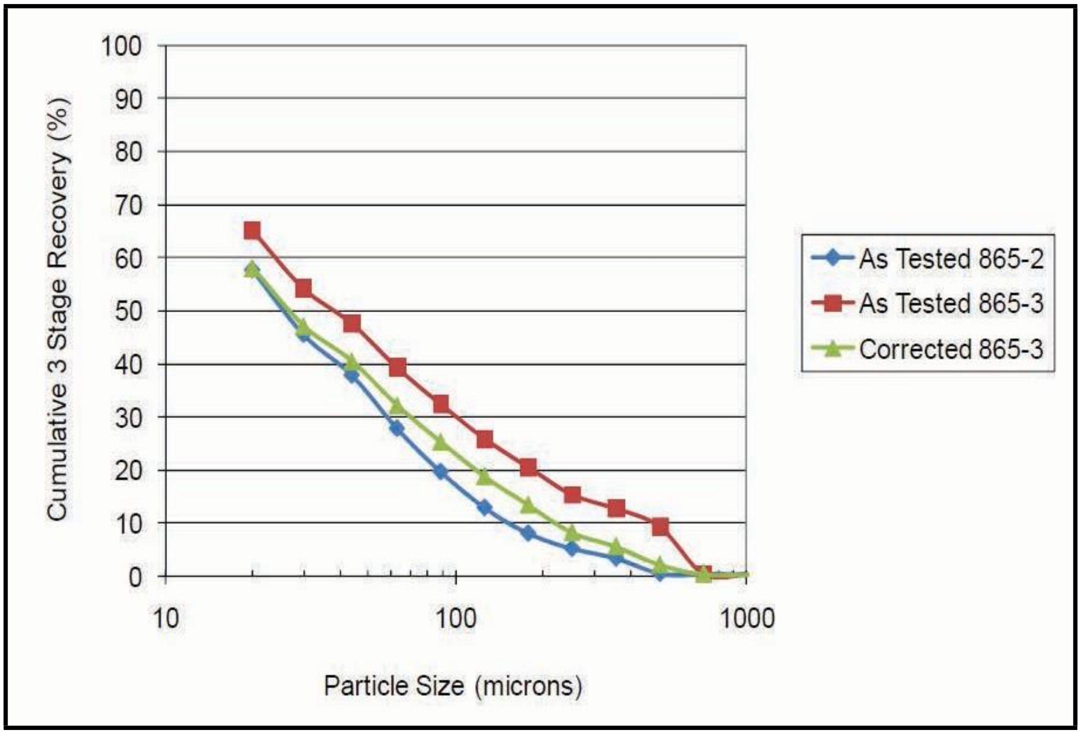


Figure 13-1 Cumulative Three-stage EGRG Results (Revised) – Knelson Research

The results indicate that there will be no significant variation in gravity recoverable gold for the two rock types tested. Modelling a potential gold recovery from a plant scale gravity circuit gold recovery using an anticipated primary grind size P_{80} of 184 µm showed a potential average gravity gold recovery of 21.3%.

Subsequent flotation test work has demonstrated that there is insignificant overall recovery difference between a combined gravity concentrate plus bulk flotation concentrate, and a bulk flotation concentrate without gravity concentration. Gravity concentration of mill feed has therefore been excluded from the anticipated process flowsheet.

13.5 Flotation

The main objective of the flotation circuit is to maximize gold recovery with concurrent TOC rejection.

Flotation parameters tested include:

- grind size
- pre-flotation
- reagent type and addition
- flotation time

- cleaning of the rougher concentrate
- rock type variation

The initial flowsheet included rougher flotation, with two stages of cleaning, to produce a product feed into a CIL circuit for gold leaching. The second cleaning stage was used to reduce the TOC, which was required to be below 1.0% and preferably below 0.5%. The flowsheet did not incorporate the recirculation of cleaner tailings to avoid a build-up of carbonaceous material, but instead tested scavenging these tailings with gravity concentration. Tests by G&T demonstrated 44.6% Au recovery from the tailings, while the test by Met-Solve recovered 55.3% of the contained gold to a combined concentrate representing 1.25% of the combined tailings stream.

13.5.1 **Grind Size**

Grind size optimization tests show that gold recovery to a rougher concentrate is not sensitive to grind size between a P_{80} 97 µm and 184 µm (see Figure 13-2 and Figure 13-3). A gold flotation recovery of 95% was achieved at the optimum grind size P_{80} of 184 µm.

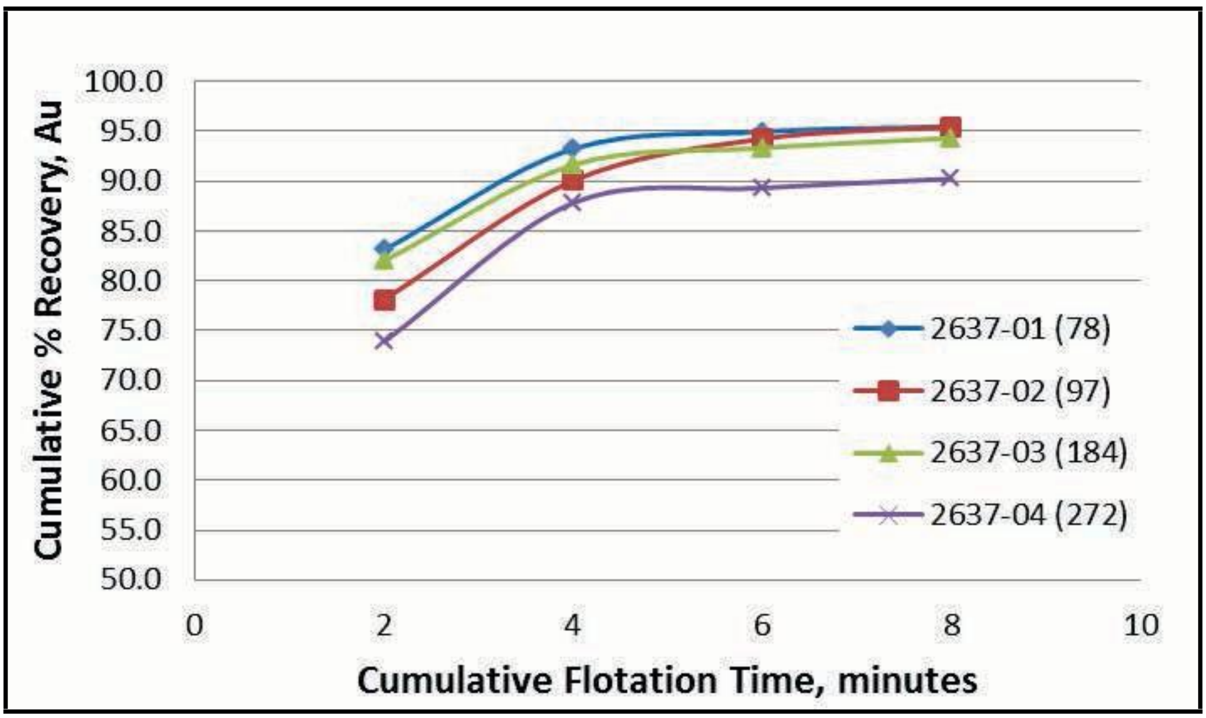


Figure 13-2 Primary Grind Size versus Flotation Kinetics – G&T

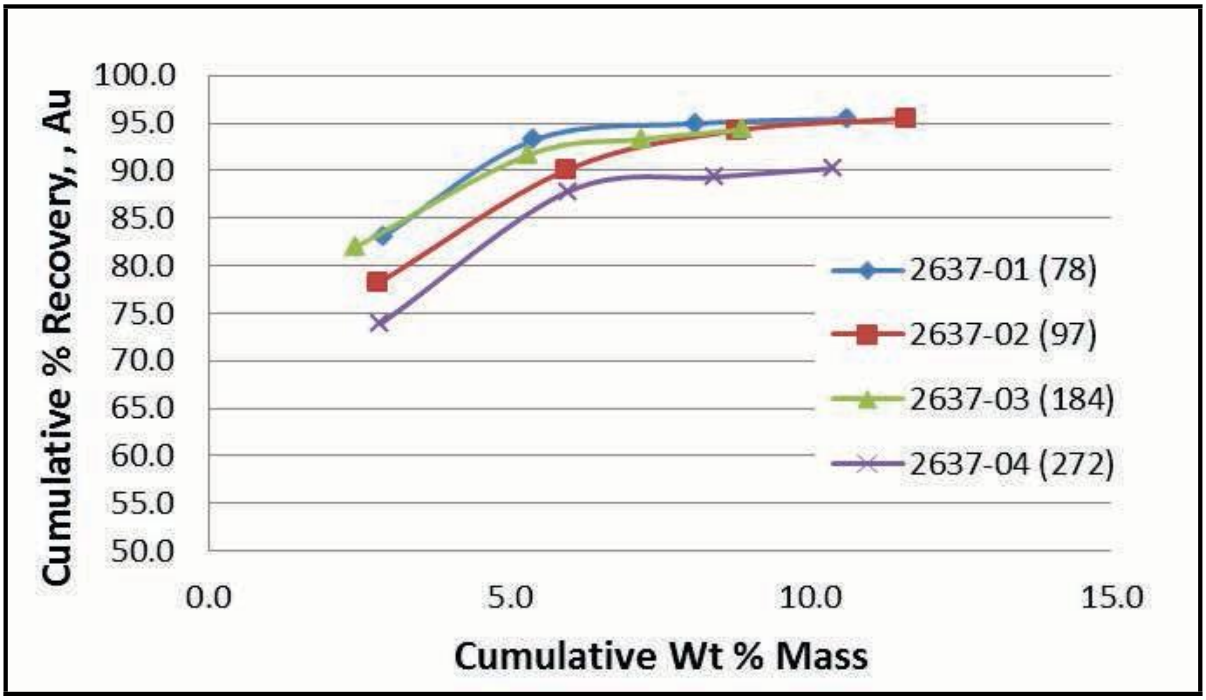


Figure 13-3 Primary Grind Size versus Mass Recovery – G&T

Open circuit rougher flotation test work at McClelland in 2019, using a P_{80} of 180 µm, achieved gold recoveries ranging from 92.9 to 94.1%, as summarized in Table 13-5.

Table 13-5 2019 Rougher Flotation results at P80 of 180 µm - McClelland

Test	Sample	Head Grade	Au Recovery
		Au g/t	%
F-1	4373-002	0.77	92.9
F-2	4373-003	1.00	93.3
F-4	4373-002	0.77	93.8
F-5	4373-003	1.00	94.1

Note: Test F-3 using composite 001 has been omitted as the head grade is too low to be considered ore.

13.5.2 Primary Flotation Reagents

Potassium amyl xanthate (PAX) has been used as a general-purpose flotation collector. Methyl isobutyl carbinol (MIBC) has been used as a frother. Carboxymethylcellulose (CMC) has been used to depress organic carbon in cleaner flotation (discussed in further detail in the following section).

13.5.3 Cleaner Flotation for Carbonaceous Material Rejection

Pre-flotation ahead of rougher flotation resulted in the reduction of TOC and reagent consumption in the subsequent flotation stages.

G&T completed tests on a sample of argillite material with a higher-than-average feed TOC content, namely 1.25% TOC, from Composite Sample 871. The test work demonstrated that the use of a pre-flotation stage with the addition of CMC to the cleaner circuit, and then the rougher circuit for graphite depression, successfully reduced TOC content to levels required for efficient leaching.

The second cleaner stage configuration adopted for the flowsheet is based on reducing the mass and upgrading the flotation concentrate product prior to regrinding and leaching. Regrinding of the rougher concentrate prior to cleaning has apparently not been tested; possibly because the anticipated generation of ultrafine carbon would interfere with the cleaner flotation process and any subsequent thickening ahead of the leaching process.

Previous test work indicated that incorporating more than one stage of cleaning had no positive impact, and possibly a negative impact, on the TOC content of the final concentrate; however, the addition of CMC decreased the TOC content of the final concentrate but required additional cleaning stages. Although the flotation circuit design is based on two stages of cleaning, it may be possible to simplify the circuit by using only one stage, particularly if column flotation or Woodgrove flotation cells are used for cleaning.

Conceptual test work at Metsolve in 2017 has successfully reduced TOC content to suitable levels using CMC in the cleaner circuit without a pre-flotation stage. The flowsheet has been subsequently adjusted to exclude pre-flotation and depress TOC in the cleaner circuit. Gravity concentration of cleaner tails was added to scavenge gold from cleaner and re-cleaner tails as shown in Figure 13-4.

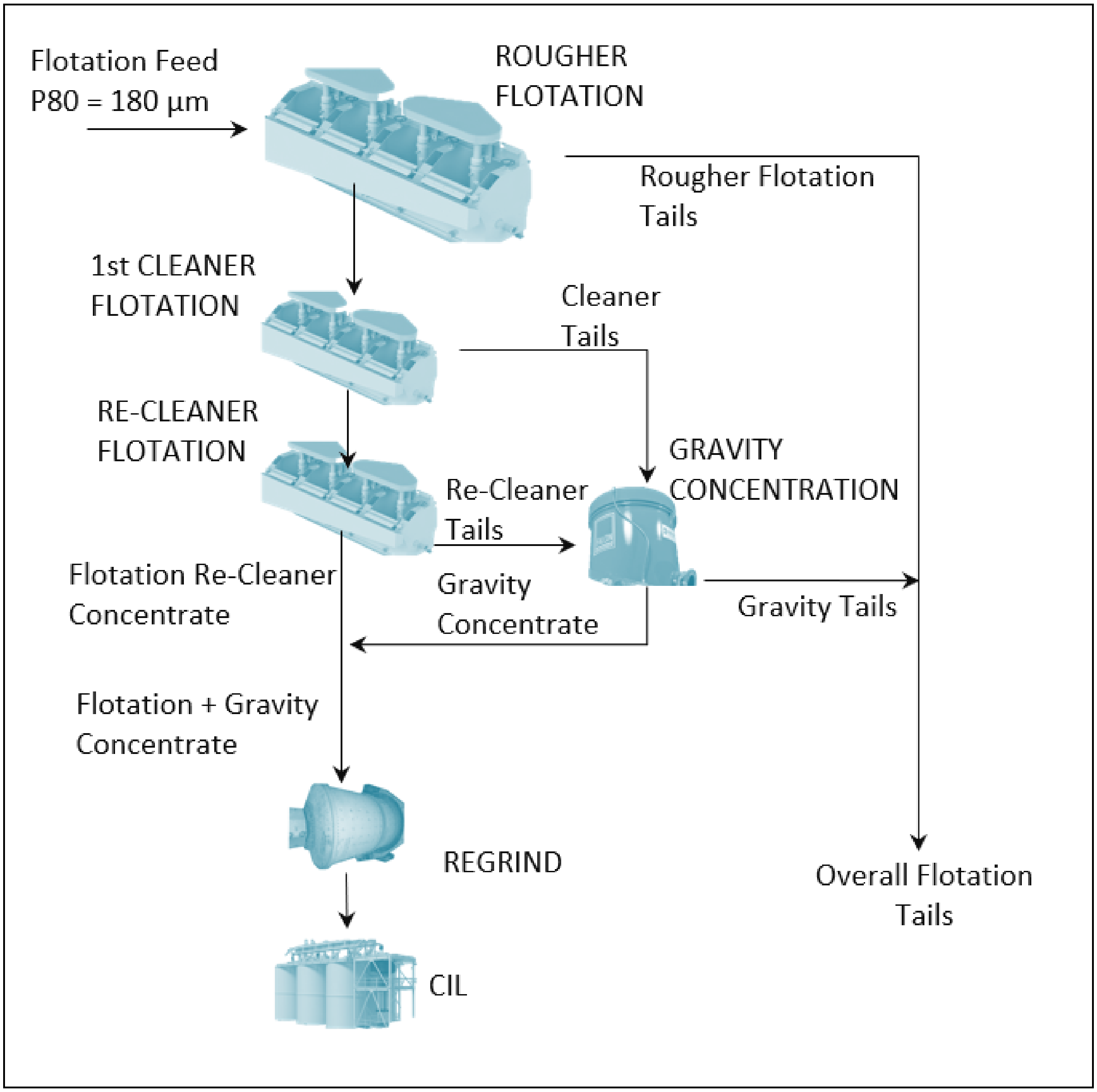


Figure 13-4 Updated Flotation and Gravity Concentration Flowsheet Overview – MMTS

13.5.4 Cleaner Flotation

Rougher flotation concentrates were combined for the cleaning test at McClelland in 2019. CMC reagent, equivalent to 0.05 kg/t ("whole ore" mass basis), was added and the slurry was conditioned for 1 minute. An additional 0.035 kg/t ("whole ore" mass basis) PAX was added, and the slurry was conditioned for another 1 minute.

The rougher concentrate was subjected to cleaner flotation, at an approximately 33% solids density (depending on rougher flotation mass pull), using a 1,200 rpm agitation rate for approximately 9 minutes. Pull time was extended, as appropriate, based on technician observations during testing. The resulting cleaner concentrate was used for recleaner flotation.

Cleaner concentrate from each test was subjected to recleaner flotation. CMC reagent, equivalent to 0.015 kg/t ("whole ore" mass basis) was added, and the slurry was conditioned for 1 minute. After 1 minute, PAX equivalent to 0.010 kg/t ("whole ore" mass basis) was added and the slurry was conditioned for another 1 minute. The cleaner concentrate was subjected to recleaner flotation, using an agitation rate of 1,200 rpm.

Flotation test results showed that the Spanish Mountain composites responded well to cleaner and recleaner flotation. Two scoping flotation tests were conducted on composite 4373-002 (18-DH-1217). Results from the two tests (F-1 and F-4) were essentially the same and showed that the composite responded very well to flotation processing. For the purposes of discussion, results from test F-1 are considered here. The recleaner concentrate produced from Comp. 4373-002 (Test F-1) was 3.21% of the ore weight, assayed 25.5 g/t Au and 32.2% sulphide sulfur, and represented a gold recovery of 91.3%. The recleaner tail was only an additional 0.13% of the ore weight, assayed an additional 5.21% of the feed weight at 0.22 g/t Au and 0.48% sulphide sulfur, and represented an additional gold recovery of only 1.3%. The combined rougher concentrate was 8.55% of the feed weight, would assay 9.74 g/t Au, and represented a gold recovery of 92.9%.

Two scoping flotation tests were conducted on composite 4373-003 (18-DH-1218). Gold recoveries from the two tests (F-2 and F-5) varied somewhat but showed that the composite responded well to flotation treatment. For the purposes of discussion, results from test that gave a better gold recovery (F-5) are considered here. The recleaner concentrate produced from Comp. 4373-003 (Test F-5) was 3.41% of the ore weight, assayed 24.5 g/t Au and 33.1% sulphide sulfur, and represented a gold recovery of 90.5%. The recleaner tail was an additional 0.37% of the ore weight, assayed 6.8 g/t Au and represented an additional gold recovery of 2.7%. The first cleaner tails were an additional 6.01% of the feed weight, assayed 0.14 g/t Au and 0.50% sulphide sulfur, and represented an additional gold recovery of only 0.9%. The combined rougher concentrate was 9.79% of the feed weight, would assay 8.89 g/t Au and represented a gold recovery of 94.1%. Although it was not possible to complete a sulphide sulfur balance for this test (insufficient sample for assay of some products), the rougher tail sulphide sulfur grade (0.12%) indicated a high sulphide sulfur recovery (94% based on a 1.70% assayed head grade). The lower gold recovery to the final (recleaner) concentrate for the other test conducted on this composite (F-2) appeared to result mainly from gold losses to the first cleaner tail from that test (4.4% of the total gold). Causes for those gold losses are not well understood.

13.5.5 Cleaner and Recleaner Tailings Scavenging

The current flowsheet includes gravity scavenging of flotation cleaner tailings. The objective of this approach is to avoid the recirculation of graphitic material from the cleaner tailings back to the rougher flotation feed. Tests carried out by G&T and Met-Solve to demonstrate the gold recovery to be achieved in this manner are summarized in
Table 13-6.

The G&T test 85 was carried out on the combined cleaner tails weighing approximately 3 kilograms from a series of batch flotation tests and the Met-Solve test was carried out on approximately 2 kilograms cleaner tailings from a single flotation test. The Met-Solve test results include hand-panning of the centrifugal concentrator concentrate. The weight % and % recovery reported in Table 13-6 are both based on the cleaner tailing representing 100%.

Table 13-6 Gravity Recovery from Flotation Cleaner Tailings - G&T and Metsolve

Test No.	Feed		Gravity Concentrate			
	Au, g/t	TOC, %	Wt. %	Au, g/t	TOC, %	Au Rec, %
KM 2637-85	1.56	0.82	5.2	13.2	0.66	44.6
ND231	0.92	0.85	0.63	72.45	-	49.4

Although the gravity concentrate may contain a higher concentration of TOC than is desirable for flotation, the very low quantity of this material, being about 1/100 of the flotation concentrate, means it can be added to the flotation concentrate for regrinding and cyanidation.

In 2019 McClelland completed flotation cleaner and re-cleaner test work followed by gravity concentration of the combined cleaner and recleaner tails. As shown in Figure 13-5, gold recovery from cleaner tails was approximately 50%, with a gravity mass pull of approximately 5%. TOC was reduced from 3.04 % in cleaner tailings to 0.44% in gravity concentrate.

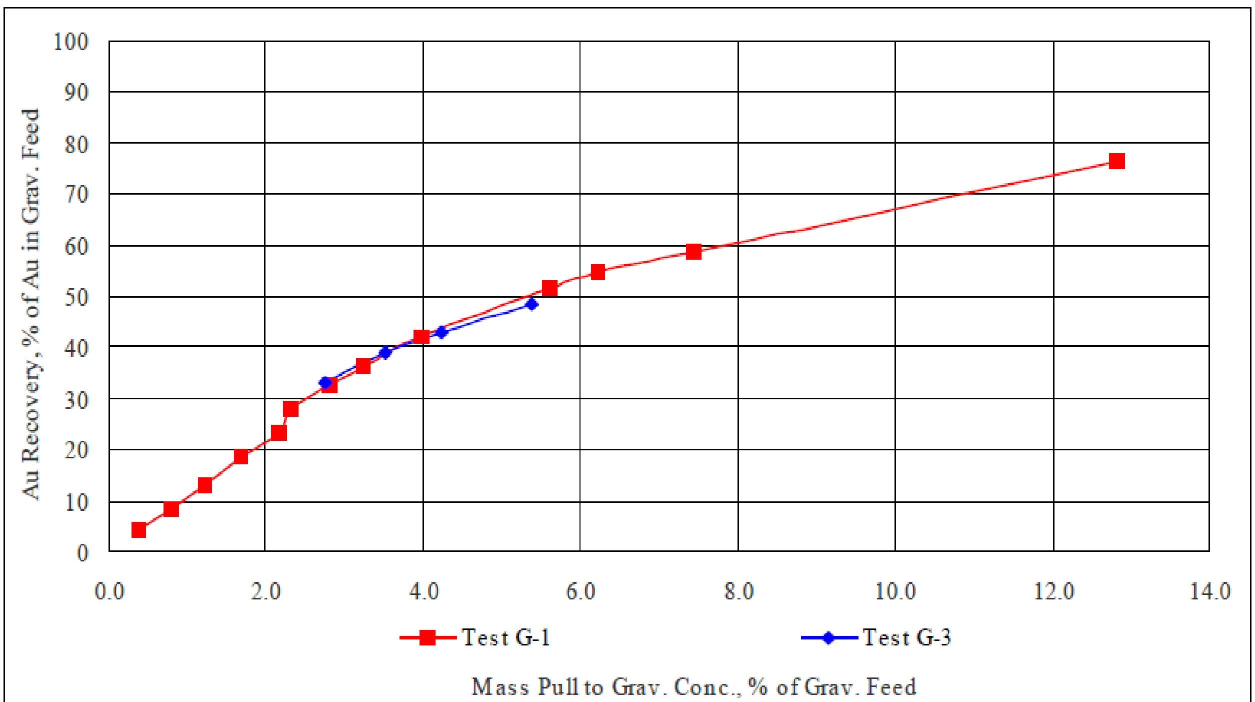


Figure 13-5 Gravity Concentration of Combined Flotation Cleaner Tails – McClelland 2019

Overall gold recovery from flotation feed to combined flotation + gravity concentrate was on average approximately 92%.

13.5.6 Regrind of Concentrate

A combined leach feed concentrate, including cleaner flotation concentrate and concentrate from gravity concentration of cleaner tails, represents a mass pull that is 3.4% of total mill feed.

Test work has established that the extraction of gold from the flotation concentrate by cyanide leaching is sensitive to the fineness of grind. A regrind P_{80} size of 20 µm is used in the current process design.

Figure 13-6 shows the gold content of flotation concentrate cyanidation tailings as a function of the regrind size of the concentrate. A finer regrind results in a lower tailings assay. Increasing gold dissolution with the increasing fineness of grind is apparent and remains particularly the case for regrind sizes of P_{80} less than 20 µm values. Additional test work to assess the potential of a regrind size P_{80} finer than 20 µm will be undertaken in future programs.

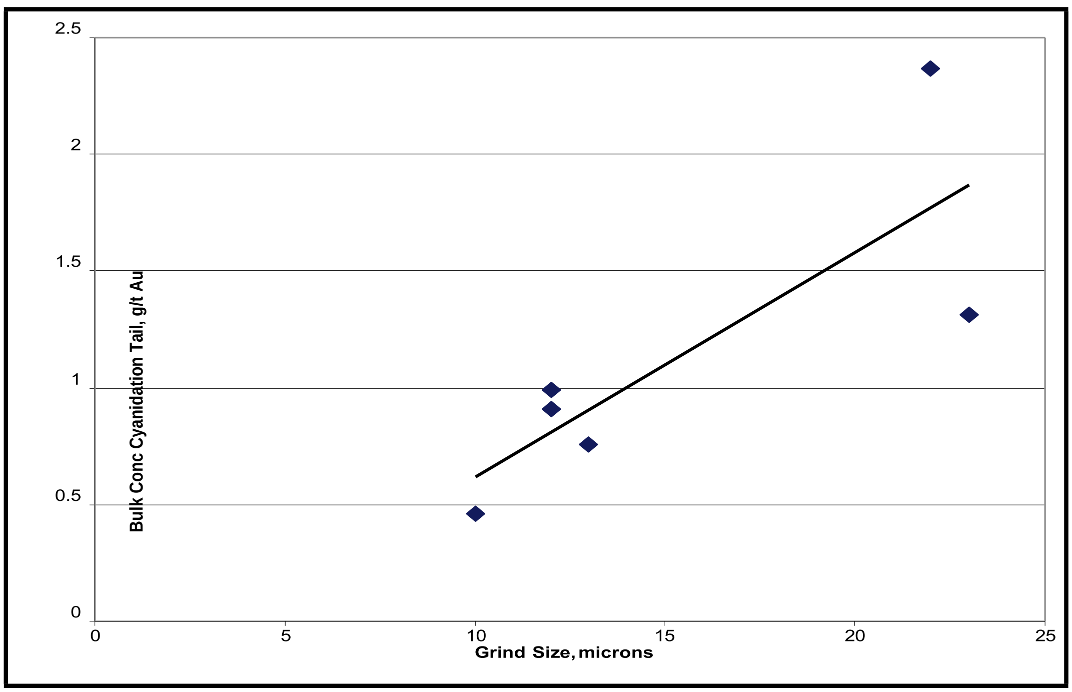


Figure 13-6 Flotation Concentrate Cyanidation Tailings versus Regrind Size – G&T

13.6 Leaching

Test work carried out by G&T during 2007 investigated the cyanide leaching of both the whole-ore sample material and of the flotation products. Various conclusions were drawn from these studies as outlined below:

- Whole-ore cyanide leaching at a primary grind P_{80} of 74 µm resulted in low recoveries.
- Direct cyanidation of flotation concentrates without a regrind prior to leaching resulted in low recoveries.
- Preg-robbing tests indicated that the samples tested displayed a natural tendency to have a very high preg-robbing activity.

- Subsequent test work and an analysis of the results obtained indicate that previous low extraction values were due to the presence of TOC, and the requirement of a fine regrind for improved gold liberation prior to leaching.

During the test work completed in 2010 and 2011, it became apparent that even with CIL leaching, control of the TOC content would be required to achieve acceptable leach recoveries of the flotation concentrate.

13.6.1 Leaching of Gravity Concentrates

Initial testing of the gravity concentrates included the use of CIL leaching for gold extraction. Gravity concentrate leaching consistently achieved a gold extraction greater than 97% from these concentrates, which generally had a TOC content of less than 0.5%.

13.6.2 CIL Leaching of Flotation Concentrates

Flotation concentrate regrind prior to leaching is essential for high gold dissolution. Although the flotation concentrate appears to benefit from a regrind size as fine as 10 µm, further investigation is required to confirm the optimal economic regrind size as previously discussed.

G&T conducted several leach tests as part of the KM2637 test program. Initial leach tests were limited to 24 hours and indicated that additional concentrate regrind, additional cyanide, and longer leach durations were required. Flotation concentrate leach tests were conducted at SGS on 16 composite samples. Concentrate slurry was pre-aerated for a variable time before being leached under CIL test procedure conditions for 48 hours. The test results obtained are summarized in Table 13-7..

Table 13-7 Flotation Concentrate Cyanidation Results – SGS

Test No.	Composite	P_{80} Grind (µm)	Leach Feed % TOC	NaCN Consumption (kg/tonne concentrate)	Au Extraction (%)
17	1	14.8	0.22	7.63	95.8
18	2	11.7	0.39	7.03	97.5
19	3	20.5	0.41	9.11	87.1
20	6	45.8	0.20	5.05	97.1
21	7	34.5	0.35	9.02	95.7
22	8	14.5	0.61	15.7	91.1
23	9	109	0.59	10.1	91.7
24	10	17.4	0.39	10.7	96.8
25	11	14.5	0.30	11.3	95.1
26	12	11.2	0.52	8.09	94.8
27	14	9.8	0.46	7.73	96.6
28	18	9.77	0.42	5.43	94.0
29	19	11.0	0.45	8.07	98.6
30	20	8.9	0.58	7.21	97.8
31	21	15.7	0.86	9.33	97.9
32	24	16.0	0.67	6.67	84.9

An analysis of the gold assays obtained from these test results indicate that the conditions required to achieve high gold extraction from the concentrate are a regrind size P_{80} of less than 20 µm and a TOC content of less than 0.5%. Average gold extraction values of about 94.5% were attained, with individual recoveries as high as 98.6%, thereby indicating that the test conditions had not been optimized. Mass pull to cleaner flotation concentrate of approximately 4%, and cyanide consumption of approximately 8.5 kg/t concentrate, results in an overall cyanide consumption of 0.34 kg/t mill feed.

G&T subjected Composite 4 material to additional tests to assess the flotation variables and leach properties of the product created. The results are shown in Table 13-8.

Table 13-8 Flotation Concentrate Variables – G&T

CIL Test No.	Primary P_{80} µm	Regrind P_{80} µm	% Au Extraction CIL	Feed % TOC	Flotation Concentrate % TOC	Pre-flotation Time (min)
44	173	12	85.1	0.80	1.48	12
47	173	14	78.9	0.80	0.80	25
59	200	23	84.8	0.80	0.96	15
65	200	11	91.5	0.83	1.08	15

Gold leach extraction of between 78.9 and 91.5% were achieved in these tests. Primary grind size and pre-flotation time did not seem to influence gold recovery. TOC recovered into the concentrate had a detrimental effect on leach recovery. Regrind P_{80} size also appears to have influenced gold extraction.

Results of additional leach test work using Composite 4 material are shown in Table 13-9. Material for these leach tests was created from a sample which had been gravity processed, included in the pre-flotation stage, and then had CMC added to the cleaner flotation stages. Except for Test 74, which had a very low TOC content prior to processing and did not have CMC added to the cleaner circuit.

Table 13-9 Composite 4 Variability Samples - G&T

Test No.	Sample No.	Regrind P_{80} µm	Flotation Feed TOC %	Concentrate TOC %	CIL Au Recovery %
93	872a	18	1.13	0.31	92.7
94	891	19	1.23	0.26	93.9
95	894	16	0.80	0.23	97.4
96	871	17	1.25	0.28	95.5
74	865-3	11	0.21	0.11	96.3

Average recovery from the flotation concentrate using the CIL-test procedure was 95%. While 24 hours appears to be an adequate leach time for non-argillite materials, materials with significant TOC content require a leach time of 48 hours.

Average gold extraction by cyanide leach was 95% in the G&T test work, and 94.5% in the SGS test work.

Combined flotation and gravity concentrates were subject to CIL tests in 2019 at McClelland and confirmed the sensitivity of gold recovery to grind size. The results summarized in Table 13-10 show that gold CIL recovery of 99% is achievable with a regrind size less than 37 µm at very low cyanide (0.1 kg/t ore) and lime consumption (0.2 kg/t ore).

Table 13-10 CIL Leach tests carried out at McClelland in 2019

Regrind Feed Size	NaCN Conc.	Au Recovery,	Reagent Requirements kg/t Ore	
µm	g/L	%	NaCN.	Lime
80% - 75 µm	2.00	90.8	0.07	0.04
100% - 37 µm	2.00	99.8	0.11	0.21
100% - 25 µm	2.00	98.8	0.1	0.1

13.7 Cyanide Destruction

Cyanide destruction test work was carried out on leach residues from the 2019 McClelland CIL tests using the SO_2/air process. The cyanide destruction tests were effective in decreasing the CIL slurry CNWAD concentration from about 500 ppm to 2 ppm, with an SO_2 addition rate equivalent to 6.0 g/g CNWAD. Copper and lime additions were not required.

13.8 Gravity Concentration Scavenging of Rougher Tailings – Future Opportunity

Scavenger gravity concentration testing was conducted on the bulk flotation rougher tailings, generated from Comp. 4373-004 in the 2019 McClelland tests, to determine gravity recoverable gold losses to the flotation tailings. Test results indicated a significant quantity of gold (48.4% of gold contained in the flotation rougher tailings) was recovered to a gravity cleaner concentrate that was 0.07% of the flotation tails weight. These results indicate that future consideration should be given to the inclusion of gravity concentration processing in the grinding circuit.

13.9 Tailings Filtration Testwork (G&T Project KM 3403)

Preliminary tests were completed on two flotation tailings samples to evaluate whether dewatering of the tailings and dry stacking could be a viable option. The tailings exhibited very poor solid-liquid separation characteristics. The addition of an anionic flocculant (Cytec 130) increased the settling rate and supernatant clarity significantly but the final density was only about 60% solids w/w. Filtration of the flocculated tailings gave very poor filtrate quality and extremely long filtration times. The net conclusion of these tests was that filtration and dry-stacking of the tailings is not a viable option and further testwork was not pursued.

13.10 Process and Metallurgical Summary

SMG gold bearing ore is generally moderate to soft. Metallurgical test work results confirm that flotation of SMG ore using CMC in cleaner flotation and gravity concentration of cleaner tails can produce a concentrate with a low TOC content to overcome potential preg-robbing properties.

Overall gold process recovery for the PEA is estimated to be 91% from a process that includes a primary grinding to a P_{80} of 184 microns, rougher flotation, flotation cleaning with CMC, gravity concentration of cleaner tails, and subsequent CIL of reground combined flotation and gravity concentrate.

Silver occurs in very minor economic proportions at SMG. Metallurgical test work indicates an estimated 27% overall silver process recovery.

There are no known additional processing factors or deleterious elements that could have a significant effect on potential economic extraction other than the factors described above. It is the opinion of the QP, Tracey Meintjes, P.Eng., that sufficient metallurgical testwork and analysis has been completed to support process design, process recovery assumptions and process cost estimates used for the Preliminary Economic Assessment.

14.0 Mineral Resource Estimate

Sue Bird, P.Eng, has been retained to produce an updated Mineral Resource Estimate ("Resource") on the Spanish Mountain Gold Deposit located approximately 6 km east of Likely, BC, and 70 km northeast of Williams Lake. Sue Bird visited the Property on September 12, 2019 to inspect the drillhole collars, drill core and security procedures as well as discuss geology and drilling practices at the site.

The Mineral Resource at the base case cut-off of 0.15 g/t Au, has an effective date of October 10, 2019, and is summarized in Table 14-1 with the sensitivity of the resource summarized in Table 14-2. Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves dated May 10, 2014 (CIM, 2014) were followed for the Mineral Resource estimate.

The resource has been confined to a "reasonable prospects of eventual economic extraction" shape based on conventional open pit mining with the following assumptions used to determine the cutoff grade:

- Gold Price = US$1,275/oz;
- Exchange Rate = 0.75 US$:1 C$;
- Process Costs (including G&A costs) = $7.25 /t;
- Process Recovery = 90%;
- Overall Slope Angles conforming to inputs listed in Table 16-5.

Additional assumptions to determine the confining pit shape are summarized in the notes below the Resource tables.

Table 14-1 Mineral Resource Estimate for Spanish Mountain at a 0.15g/t Au Cut-off

Classification	Tonnage	Grade		Contained Metal	
	Mt	Au, g/t	Ag, g/t	Au, koz.	Ag, koz.
Measured	29.6	0.60	0.83	569	791
Indicated	243.6	0.46	0.69	3,566	5,413
Measured + Indicated	**273.2**	**0.47**	**0.71**	**4,135**	**6,204**
Inferred	52.4	0.37	0.67	619	1,128

Table 14-2 Sensitivity of Mineral Resource to Au Cut-Off Grade (Base Case Highlighted)

Classification	Cut-off Grade	Tonnage	Grade		Contained Metal	
	Au (g/t)	Mt	Au, g/t	Ag, g/t	Au, koz.	Ag, koz.
Measured	**0.15**	**29.6**	**0.60**	**0.83**	**569**	**791**
	0.30	19.2	0.80	0.80	496	491
	0.40	14.6	0.95	0.80	446	376
	0.50	11.3	1.10	0.81	397	293
	0.60	8.8	1.25	0.82	355	234
	0.70	7.1	1.40	0.84	318	192
Indicated	**0.15**	**243.6**	**0.46**	**0.69**	**3,566**	**5,413**
	0.30	123.1	0.69	0.73	2,735	2,894
	0.40	82.5	0.86	0.76	2,284	2,022
	0.50	58.9	1.03	0.79	1,946	1,491
	0.60	43.4	1.20	0.80	1,673	1,121
	0.70	32.6	1.38	0.83	1,449	868
Measured + Indicated	**0.15**	**273.2**	**0.47**	**0.71**	**4,135**	**6,204**
	0.30	142.3	0.71	0.74	3,231	3,385
	0.40	97.1	0.87	0.77	2,729	2,398
	0.50	70.2	1.04	0.79	2,343	1,784
	0.60	52.2	1.21	0.81	2,028	1,355
	0.70	39.7	1.39	0.83	1,768	1,059
Inferred	**0.15**	**52.4**	**0.37**	**0.67**	**619**	**1,128**
	0.30	20.6	0.61	0.69	401	456
	0.40	12.5	0.78	0.72	312	288
	0.50	8.0	0.96	0.72	247	184
	0.60	4.9	1.23	0.75	192	117
	0.70	2.8	1.64	0.82	150	75

Notes for Resource Tables:

- *Mineral Resources have an effective date of October 10, 2019 and are prepared in accordance with CIM Definition Standards and NI 43-101. The Qualified Person for the estimate is Sue Bird, P.Eng.*
- *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.*
- *Inferred Resources are not based on sufficient drilling to be considered Measured or Indicated and it is not certain that further exploration will result in upgrading the classification. As such, Inferred resources have not been used in the mine plan.*
- *Silver value is not considered in the cut-off grade estimation.*
- *Considerations for the Lerchs-Grossman algorithm used to define the "reasonable prospects of eventual economic extraction" open pit shell are the same as those listed above for the cutoff grade determination, as well as a $2.20/t mining cost. Overall pit slope angles range from 20 degrees to 43 degrees and are estimated based on geotechnical analysis of various zones in the deposit.*

Factors that may affect the estimates include: metal price assumptions, changes in interpretations of mineralization geometry and continuity of mineralization zones, changes to kriging assumptions, metallurgical recovery assumptions, operating cost assumptions, confidence in the modifying factors, including assumptions that surface rights to allow mining infrastructure to be constructed will be forthcoming, delays or other issues in reaching agreements with local or regulatory authorities and stakeholders, and changes in land tenure requirements or in permitting requirement.

The following Section summarizes the data, analyses, interpretations, interpolations and validations used to produce the block model for the Spanish Mountain deposit Resource Estimate.

14.1 Resource Estimation Procedure

The resource evaluation methodology involved the following procedures:

- Database compilation and verification;
- Construction of overburden and boundaries of the mineralization;
- Definition of resource domains;
- Compositing and outlier restriction analyses;
- Geostatistical analysis and variography;
- Block modelling and grade interpolation;
- Resource classification and validation;
- Assessment of "reasonable prospects for economic extraction" and selection of appropriate cut-off grades

14.2 Resource Database

In total, 852 drillholes are within the block model volume and have been drilled since 2004. Drilling prior to this date has not been used because both the core and the certificates are not available and therefore there was no method to validate the data. A summary of the drilling by year is provided in Table 14-3.

A summary of drilling used for the modelling, by drill type and size is provided in Table 14-4. Figure 14-1 illustrates the drilling by type in plan with the resource pit shown as well.

Table 14-3 Summary of Drillhole Data used in Resource Estimate by Year

	DH Data within the Block Model Limits					% of Total			
Year	# Holes Drilled	Total Length drilled (m)	# of Assay Intervals	Total Length Assayed (m)	% Assayed	# Holes Drilled	Total Length drilled (m)	# of Assay Intervals	Total Length Assayed (m)
2004	32	2,405	1,501	2,285	95%	4%	1%	1%	1%
2005	65	11,123	7,583	10,724	96%	8%	6%	7%	6%
2006	115	24,886	16,616	24,019	97%	13%	14%	14%	14%
2007	106	23,894	15,619	22,673	95%	12%	13%	14%	13%
2008	158	39,661	25,879	38,508	97%	19%	22%	22%	22%
2009	57	12,475	8,137	11,983	96%	7%	7%	7%	7%
2010	20	6,835	4,059	6,325	93%	2%	4%	4%	4%
2011	82	19,436	13,077	18,985	98%	10%	11%	11%	11%
2012	132	24,336	14,800	23,401	96%	15%	14%	13%	14%
2013	56	9,229	6,010	9,032	98%	7%	5%	5%	5%
2014	18	2,676	1,684	2,531	95%	2%	2%	1%	1%
2018	11	1,091	717	1,091	100%	1%	1%	1%	1%
Total	**852**	**178,047**	**115,682**	**171,558**	**96%**	**100%**	**100%**	**100%**	**100%**

Table 14-4 Summary of Drillhole Data by Type of Drilling

	# Holes by Type				Assayed Length by Type				% of Assayed Length			
Year	**NQ**	**HQ**	**RC**	**# Holes**	**NQ**	**HQ**	**RC**	**Assayed Length**	**NQ**	**HQ**	**RC**	**Assayed Length**
2004	0	0	32	32			2,285	2,285	0%	0%	11%	1%
2005	35	0	30	65	7,509		3,214	10,724	5%	0%	15%	6%
2006	88	0	27	115	21,253		2,765	24,019	15%	0%	13%	14%
2007	106	0	0	106	22,673			22,673	16%	0%	0%	13%
2008	158	0	0	158	38,508			38,508	27%	0%	0%	22%
2009	26	30	0	57	6,771	5,212		11,983	5%	69%	0%	7%
2010	8	12	0	20	3,984	2,341		6,325	3%	31%	0%	4%
2011	82	0	0	82	18,985			18,985	13%	0%	0%	11%
2012	132	0	0	132	23,401			23,401	16%	0%	0%	14%
2013	0	0	56	56			9,032	9,032	0%	0%	43%	5%
2014	0	0	18	18			2,531	2,531	0%	0%	12%	1%
2018	0	0	11	11			1,091	1,091	0%	0%	5%	1%
Grand Total	**635**	**42**	**174**	**852**	**143,085**	**7,553**	**20,919**	**171,558**	**100%**	**100%**	**100%**	**100%**
% of Each Drill Type	75%	5%	20%		83%	4%	12%					

Missing or un-sampled intervals were filled with 0.001 g/t Au. Samples not sampled for silver were left blank.

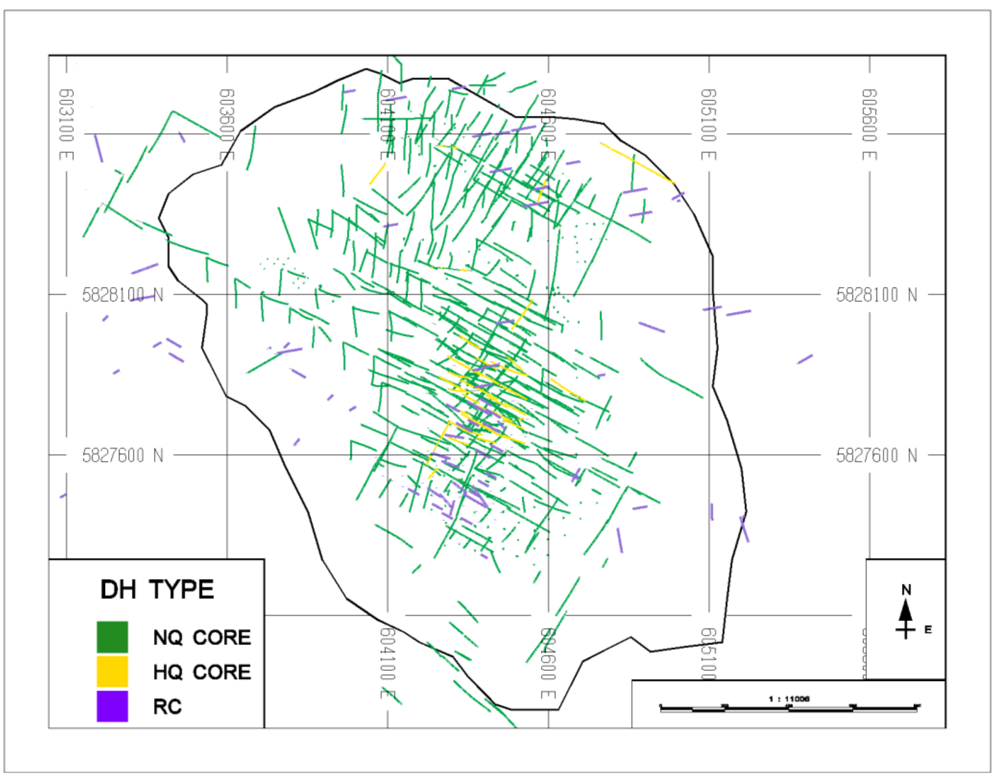


Figure 14-1 3D View of Drillholes by DH Type with the Spanish Mountain Resource Shell outline

14.2.1 Drillhole Type Comparisons

As illustrated above, there has been NQ core, HQ core and RC drilling at the site. A comparison of Au grades for each type has been completed using Point Validation or "bootstrapping" to compare the composites in the immediate vicinity of the data to be compared.

For the RC drilling comparison, composites from NQ and HQ core were used to interpolate to the locations of the RC drilling. For the HQ comparison only, NQ core was used. To ensure that only nearby data is compared, only the first pass of the 4-pass interpolation (as employed for the grade modelling) has been used resulting in maximum distances of between 16 m and 50 m to the composites, depending on the domain. The results of this Point Validation indicate that RC drilling has mean grades 27% higher than NQ and HQ, with negligible difference in HQ and surrounding drilling.

Reasons for this difference in grades could be due to sample size difference or to other factors such as RC drilling below the water table.

Table 14-5 Summary of Point Validation Results - Mean Au Grades by DH Type

Parameter	RC		HQ	
	AU-ACTUAL	AU-INTERPOLATED	AU-ACTUAL	AU-INTERPOLATED
Num Samples	3172	3172	1436	1436
Num Missing Samples	0	0	0	0
Min	0.001	0.001	0.001	0.001
Max	26.519	11.574	67.583	5.681
Weighted mean	0.4431	0.3227	0.5414	0.5396
1-Interpolated/Actual	**27%**		**0.3%**	
% of Data within Block Model	12%		4%	

14.3 Geologic Model

A three-dimensional geologic model was originally produced by SMG for the 2017 resource update (SMG, 2017). The main zone mineralization was modelled into Upper Argillite, Altered Siltstone, Tuff and Lower Argillite with the North Zone Argillite a separate solid. All material, outside of these domains, is considered waste rock. The model has been checked with the new drilling and minor adjustments were made.

The overburden surface has been used to clip all lithologies and domains. Figure 14-2 illustrates the lithology model. The lithology solids have been made slightly transparent in order to also show the drillhole location by year drilled. The Northing and Easting gridlines indicate the scale as well as the limits of the block model used for the Resource estimate. The Claims Boundary shown in this Figure illustrates that the entire 3D block model is within the SMG Claims.

Due to a significant change in dip of the three main mineralized lithologies, domains have been created to separate the flatter dipping area in the south from the steeper dipping section to the north, as illustrated in Figure 14-3.

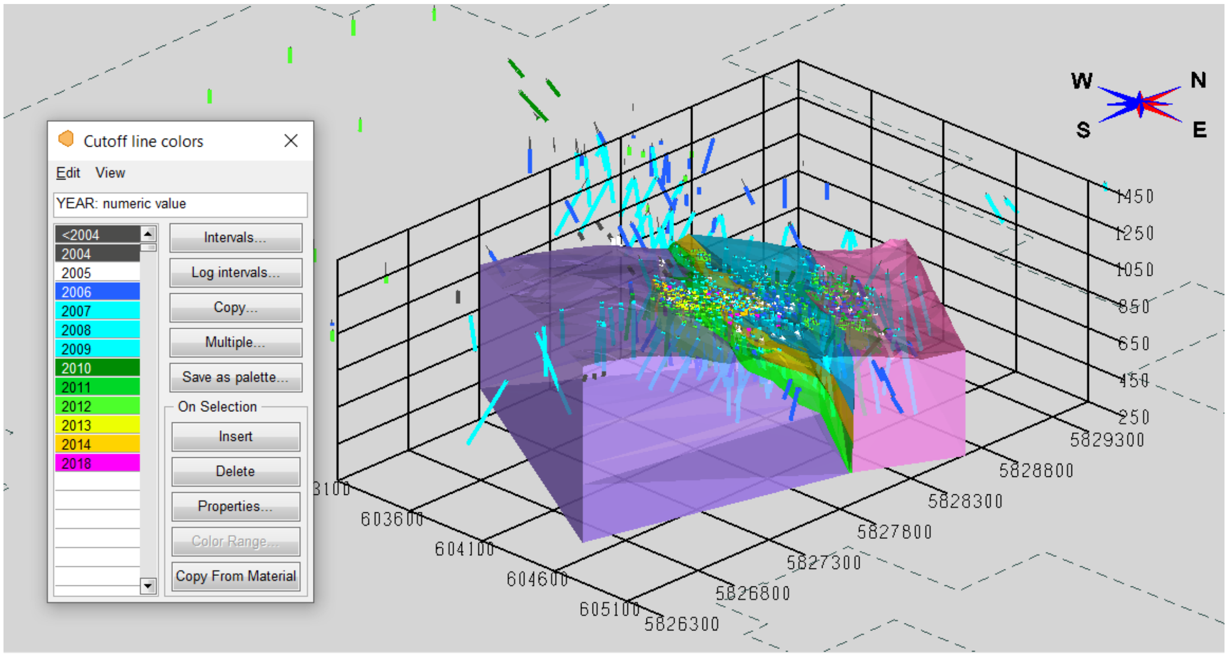


Figure 14-2 3d View Looking Az=315deg. Lithology as: Lower Argillite-purple, Tuff-green, Upper Argillite-yellow, Siltstone-blue and North Zone Argillite-pink. Claims Boundary – dotted line.

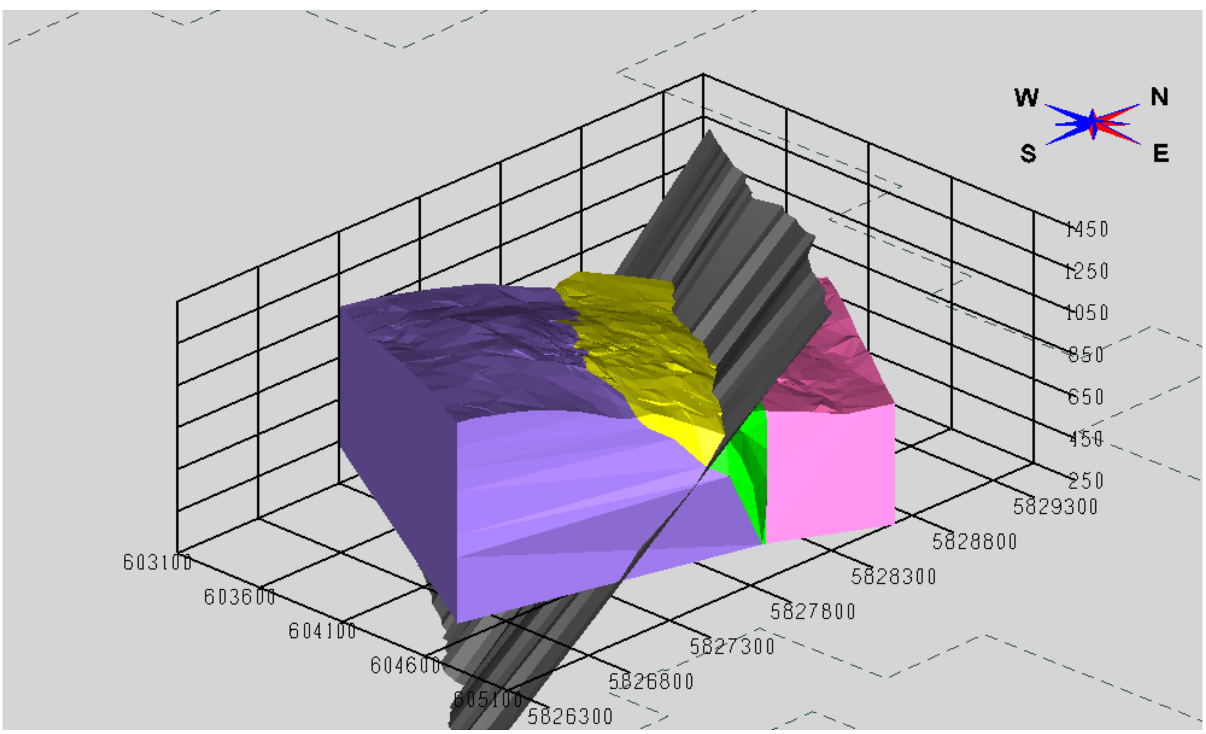


Figure 14-3 3d View Looking Az=315deg. Domains as: Lower Argillite-purple, Tuff, Upper Argillite, and Siltstone – South in yellow, North in green and North Zone Argillite-pink. Plane of Inflection in Black and Claims Boundary – dotted line.

14.4 Compositing and Outlier Restriction

Compositing has been done honouring the domain boundaries at 2.5 m fixed length intervals. Table 14-6 and Table 14-7 summarize the composite statistics for Au and Ag respectively by domain and compare them to the original assays, illustrating that the weighted mean grades remain very close after compositing. The Coefficients of Variation (C.V.) of the composites for Au range from 2.2-6.3 and are therefore considered too high to warrant linear interpolation methods. Therefore multiple indicator kriging (MIK) has been used for Au interpolations of each domain. The Ag composited C.V.s are within a range to allow for linear interpolation methods.

Table 14-6 Comparison of Assay and Composites Statistics - Au

Source	Parameter	UARG-S	UARG-N	TUFF-S	TUFF-N	LARG	SLTST-S	SLTST-N	NZ-ARG
Assays	Num Samples	12,499	3,308	19,129	4,382	42,679	3,730	6,525	18,088
	Num Missing	52	2	53	1	46	15	9	32
	Min	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001
	Max	83.400	73.900	225.000	106.000	241.000	20.000	39.000	54.400
	Wtd mean	0.504	0.306	0.317	0.216	0.213	0.060	0.066	0.240
	Weighted CV	2.751	5.735	6.717	7.054	7.970	8.732	10.579	2.780
Comps	Num Samples	7,333	1,992	11,163	2,573	25,426	2,210	3,910	10,972
	Num Missing	189	0	111	10	80	76	52	48
	Min	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001
	Max	50.441	73.611	67.659	32.012	125.459	10.748	15.086	16.671
	Wtd mean	0.505	0.306	0.317	0.216	0.213	0.060	0.066	0.240
	Weighted CV	2.159	5.602	4.245	4.396	5.558	6.292	6.323	1.987
Difference in Wtd. Mean		**0.1%**	**0.0%**	**-0.1%**	**0.0%**	**0.0%**	**0.0%**	**0.0%**	**0.0%**

Table 14-7 Comparison of Assay and Composites Statistics - Ag

Source	Parameter	UARG-S	UARG-N	TUFF-S	TUFF-N	LARG	SLTST-S	SLTST-N	NZ-ARG
Assays	Num Samples	11,150	3,280	16,486	4,277	36,037	3,594	6,360	17,813
	Num Missing	0	0	0	0	0	0	0	0
	Min	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1
	Max	88.900	9.300	84.100	63.300	30.000	25.300	28.200	103.000
	Weighted mean	0.770	1.203	0.426	0.460	0.597	0.417	0.389	0.653
	Weighted CV	1.755	0.887	2.361	2.408	1.255	1.627	1.510	1.923
Comps	Num Samples	6,544	1,983	9,760	2,519	21,593	2,172	3,875	10,880
	Num Missing	0	0	0	0	0	0	0	0
	Min	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1
	Max	53.440	8.378	29.700	19.800	13.389	13.961	13.388	35.240
	Weighted mean	0.766	1.200	0.426	0.460	0.595	0.419	0.391	0.653
	Weighted CV	1.462	0.833	1.676	1.659	1.057	1.325	1.245	1.410
Difference in Wtd. Mean		**-0.5%**	**-0.2%**	**0.2%**	**-0.2%**	**-0.4%**	**0.4%**	**0.5%**	**0.0%**

Restriction of outlier grades for MIK interpolations of Au has been applied by adjustment of the pre-defined means for the upper grade indicators.

Outlier restriction to the Ag interpolation was applied during interpolation to restrict the influence of high grade outliers. The outlier values have been determined based on Cumulative Probability Plots (CPPs) of metals in each domain.

Tabulated below are the outlier restrictions that were applied to the Ag grade interpolations. Values above the threshold were not used in interpolation of any block whose centroid is further from the composite than the distance listed in the Table.

Table 14-8 Summary of Outlier Restriction by Domain - Ag

DOMAIN	DOMAIN #	OUTLIER CUTOFF	MAX SEARCH DISTANCE (m)
UARG-S	1	10	5
UARG-N	2	5	10
TUFF-S	3	2	20
TUFF-N	4	3.5	10
LARG	5	12	10
SLTST-S	6	5	10
SLTST-N	7	2	20
NZ-ARG	8	15	10

14.5 Variography

Variograms have been created for all Indicator bins and for each domain for Au and for each domain for Ag. The orientation of the variography remains the same for each Au grade bin and for Ag as summarized in Table 14-9.

Cut-off bins for Au have been established so that each bin contains approximately the same Au metal content.

Correlogram parameters for Au are summarized in Table 14-10 and Table 14-11. Correlograms for Ag are summarized in Table 14-12.

Table 14-9 Rotation Parameters by Domain – Au and Ag

Domain	ROT-Z	ROT-X	ROT-Y
1	29	-14	0
2	29	-60	0
3	29	-14	0
4	29	-60	0
5	10	-37	0
6	30	-25	0
7	30	-62	0
8	25	-43	0

Table 14-10 Summary of Correlogram Parameters for Gold Domains 1-5

						Ranges - Spherical1			Ranges - Spherical2		
Dom	**Ind**	**Cut-off**	**C0**	**C1**	**C2**	**Y ("Major")**	**X ("Minor")**	**Z ("Vert")**	**Y ("Major")**	**X ("Minor")**	**Z ("Vert")**
1	1	0.15	0.4	0.4	0.3	25	12	15	160	140	70
1	2	0.4	0.4	0.4	0.3	18	12	15	160	140	70
1	3	0.7	0.4	0.4	0.3	15	10	15	120	100	60
1	4	1.1	0.4	0.4	0.2	10	10	12	110	80	60
1	5	1.6	0.6	0.2	0.2	10	10	10	80	80	45
1	6	2.2	0.7	0.2	0.1	20	15	15	60	40	35
1	7	3.5	0.75	0.2	0.1	15	15	6	25	30	8
2	1	0.15	0.4	0.3	0.4	8	8	10	45	120	60
2	2	0.2	0.4	0.4	0.3	8	8	10	40	90	40
2	3	0.3	0.4	0.3	0.3	8	8	10	40	60	30
2	4	0.59	0.4	0.3	0.3	8	7	8	30	40	25
2	5	0.92	0.55	0.3	0.2	8	7	8	20	30	20
2	6	1.58	0.75	0.2	0.1	6	6	6	15	20	15
2	7	13.64	0.9	0.1	0.1	5	5	6	12	15	12
3	1	0.15	0.4	0.4	0.3	25	12	20	140	120	80
3	2	0.4	0.45	0.4	0.2	18	12	20	140	85	80
3	3	0.7	0.55	0.4	0.1	15	15	15	120	50	50
3	4	1.2	0.6	0.3	0.1	12	8	10	55	45	40
3	5	2.1	0.7	0.2	0.2	10	6	8	35	40	25
3	6	4.4	0.8	0.2		15	15	6			
3	7	11.3	0.9	0.1		10	8	2			
4	1	0.15	0.45	0.4	0.2	18	20	10	50	50	50
4	2	0.3	0.5	0.4	0.2	15	12	10	40	30	20
4	3	0.5	0.55	0.3	0.2	15	8	10	40	20	20
4	4	0.89	0.6	0.3	0.2	10	8	8	20	15	15
4	5	1.5	0.75	0.2	0.1	8	6	5	15	12	10
4	6	3.1	0.8	0.1	0.1	6	6	5	10	10	8
4	7	7.28	0.9	0.1	0.1	4	4	2	7	7	5
5	1	0.15	0.4	0.4	0.2	13	15	20	110	120	100
5	2	0.3	0.45	0.3	0.3	8	8	8	70	85	60
5	3	0.5	0.45	0.4	0.2	8	7	8	65	70	50
5	4	0.8	0.52	0.3	0.2	8	7	8	55	50	50
5	5	1.4	0.65	0.3	0.1	10	7	8	50	40	30
5	6	2.5	0.75	0.2	0.1	10	5	10	25	20	20
5	7	8.4	0.8	0.2	0.1	5	4	10	10	10	20

Table 14-11 Summary of Correlogram Parameters for Gold Domains 6-8

						Ranges - Spherical1			Ranges - Spherical2		
Dom	Ind	Cut-off	C0	C1	C2	Y ("Major")	X ("Minor")	Z ("Vert")	Y ("Major")	X ("Minor")	Z ("Vert")
6	1	0.15	0.45	0.4	0.2	15	25	10	60	60	20
6	2	0.19	0.5	0.4	0.2	10	15	5	40	45	20
6	3	0.39	0.6	0.3	0.1	8	10	5	10	15	15
6	4	0.71	0.7	0.2	0.1	8	8	5	10	15	15
6	5	2.7	0.75	0.2	0.1	7	7	5	10	12	12
6	6	7.4	0.8	0.2	0.1	4	4	4	8	8	8
6	7	8	0.9	0.1	0.1	2	2	2	5	5	5
7	1	0.15	0.45	0.4	0.2	20	25	20	40	80	40
7	2	0.3	0.45	0.4	0.2	20	25	15	35	60	30
7	3	0.68	0.55	0.4	0.1	20	20	10	25	30	25
7	4	1.38	0.6	0.3	0.1	15	15	8	20	20	15
7	5	4.05	0.7	0.2	0.1	8	10	5	10	15	10
7	6	5.68	0.8	0.2	0.1	5	8	4	8	10	7
7	7	10.33	0.9	0.1	0.1	*3*	4	*3*	*6*	8	*5*
8	1	0.15	0.4	0.4	0.3	20	25	8	100	110	100
8	2	0.3	0.4	0.4	0.3	20	25	6	80	100	50
8	3	0.4	0.45	0.4	0.2	20	20	6	70	100	50
8	4	0.6	0.5	0.4	0.2	20	20	6	55	90	45
8	5	0.8	0.52	0.3	0.2	20	20	6	50	80	45
8	6	1.19	0.62	0.3	0.1	20	20	6	45	75	30
8	7	2.1	0.7	0.2	0.1	20	20	6	25	30	12

Table 14-12 Summary of Correlogram Parameters for Silver

				Ranges - Spherical1			Ranges - Spherical2		
Dom	C0	C1	C2	Y ("Major")	X ("Minor")	Z ("Vert")	Y ("Major")	X ("Minor")	Z ("Vert")
1	0.3	0.4	0.3	25	50	8	120	135	35
2	0.2	0.3	0.5	25	30	30	60	130	40
3	0.8	0.2		20	8	30			
4	0.7	0.3		25	30	30			
5	0.3	0.5	0.2	30	30	33	110	150	115
6	0.6	0.25	0.15	15	40	30	90	80	75
7	0.5	0.4	0.1	15	30	20	80	50	40
8	0.5	0.35	0.15	25	40	30	100	125	200

14.6 Block Model

The block model dimensions, and block size, are summarized in Table 14-13. The block model is a multiple percent model with 2 zones per block based on the percentage of each domain and/or overburden within the block. Total block values are then calculated as the weighted average of the grades in each domain in the block.

Table 14-13 Block Model Dimensions

Direction	Minimum	Maximum	Length (m)	Block Size	# Blocks
Easting	603,125	605,375	2,250	15	150
Northing	5,826,305	5,829,560	3,255	15	217
Elevation	245	1,450	1,205	5	241

14.7 Bulk Density

Blocks within the block model were assigned a specific gravity based on lithology as summarized in Table 14-14. The bulk density of the tuff has been adjusted to account for a porosity of about 12%. A bulk density of 2.1 was assumed for overburden. Blocks are assigned the weighted average sg of each domain within the block.

Table 14-14 Summary of Specific Gravity by Lithology

Lithology	SG
Upper Argillite	2.76
Tuff	2.46
Lower Argillite	2.76
Siltstones	2.78
North Zone Argillite	2.77
Waste	2.77

14.8 Grade Interpolation

Grades for Au were interpolated by multiple indicator kriging (MIK) due to the high C.V. values for Au domains, as discussed above. Grades for Ag were interpolated using inversed distance cubed (ID3). The rotation parameters for both Au and Ag are the same as the variogram rotation parameters, as summarized in Table 14-9. Total ellipsoidal search distances using anisotropic distances for Au and Ag are summarized in Table 14-15 through Table 14-17.

The interpolations for both Au and Ag have been done in 4 passes to limit smoothing with Pass 3 equal to the (Total Variogram Model Range) * 1.25, Pass1 = Pass3 * 0.25, Pass2 = Pass3 * 0.5 and a final Pass4 to (Total Variogram Model Range) * 2.

Additional restrictions during interpolation on composites used are summarized in Table 14-18. These parameters ensure that at least two drillholes are used for each pass of the interpolations.

Table 14-15 Summary of Search Distance Parameters for Pass 3 – Au – Domains 1-5

			Search Distance		
Dom	**Ind**	**Cut-off**	**Y ("Major")**	**X ("Minor")**	**Z ("Vert")**
1	1-2	0.15	200	175	87.5
1	2	0.4	200	175	87.5
1	3	0.7	150	125	75
1	4	1.1	137.5	100	75
1	5	1.6	100	100	56.25
1	6	2.2	75	50	43.75
1	7	3.5	31.25	37.5	10
2	1-2	0.15	56.25	150	75
2	2	0.2	50	112.5	50
2	3	0.3	50	75	37.5
2	4	0.59	37.5	50	31.25
2	5	0.92	25	37.5	25
2	6	1.58	18.75	25	18.75
2	7	13.64	15	18.75	15
3	1-2	0.15	175	150	100
3	2	0.4	175	106.25	100
3	3	0.7	150	62.5	62.5
3	4	1.2	68.75	56.25	50
3	5	2.1	43.75	50	31.25
3	6	4.4	18.75	18.75	7.5
3	7	11.3	12.5	10	2.5
4	1-2	0.15	62.5	62.5	62.5
4	2	0.3	50	37.5	25
4	3	0.5	50	25	25
4	4	0.89	25	18.75	18.75
4	5	1.5	18.75	15	12.5
4	6	3.1	12.5	12.5	10
4	7	7.28	8.75	8.75	6.25
5	1-2	0.15	137.5	150	125
5	2	0.3	87.5	106.25	75
5	3	0.5	81.25	87.5	62.5
5	4	0.8	68.75	62.5	62.5
5	5	1.4	62.5	50	37.5
5	6	2.5	31.25	25	25
5	7	8.4	12.5	12.5	25

Table 14-16 Summary of Search Distance Parameters for Pass 3 – Au – Domains 6 - 8

Dom	Ind	Cut-off	Search Distance Y ("Major")	Search Distance X ("Minor")	Search Distance Z ("Vert")	Search Distance Y ("Major")	Search Distance X ("Minor")	Search Distance Z ("Vert")
6	1-2	0.15	75	75	25	75	75	25
6	2	0.19	50	56.25	25	50	56.25	25
6	3	0.39	12.5	18.75	18.75	12.5	18.75	18.75
6	4	0.71	12.5	18.75	18.75	12.5	18.75	18.75
6	5	2.7	12.5	15	15	12.5	15	15
6	6	7.4	10	10	10	10	10	10
6	7	8	6.25	6.25	6.25	6.25	6.25	6.25
7	1-2	0.15	50	100	50	50	100	50
7	2	0.3	43.75	75	37.5	43.75	75	37.5
7	3	0.68	31.25	37.5	31.25	31.25	37.5	31.25
7	4	1.38	25	25	18.75	25	25	18.75
7	5	4.05	12.5	18.75	12.5	12.5	18.75	12.5
7	6	5.68	10	12.5	8.75	10	12.5	8.75
7	7	10.33	7.5	10	6.25	7.5	10	6.25
8	1-2	0.15	125	137.5	125	125	137.5	125
8	2	0.3	100	125	62.5	100	125	62.5
8	3	0.4	87.5	125	62.5	87.5	125	62.5
8	4	0.6	68.75	112.5	56.25	68.75	112.5	56.25
8	5	0.8	62.5	100	56.25	62.5	100	56.25
8	6	1.19	56.25	93.75	37.5	56.25	93.75	37.5
8	7	2.1	31.25	37.5	15	31.25	37.5	15

Table 14-17 Summary of Search Distance Parameters for Pass 3 - Ag

Dom	Y ("Major")	X ("Minor")	Z ("Vert")
1	150	168.75	43.75
2	75	162.5	50
3	25	10	37.5
4	31.25	37.5	37.5
5	137.5	187.5	143.75
6	112.5	100	93.75
7	100	62.5	50
8	125	156.25	250

Table 14-18 Summary of Composite Restrictions

Parameter	Pass			
	1	2	3	4
Minimum # Comps	4	4	4	2
Maximum # Comps	8	8	8	6
Max / DH	3	3	3	3

14.9 Model Validation

The modelling methods, outlier restriction of Ag, pre-defined mean grades for MIK interpolation of Au, and search parameters have been chosen so that the final interpolated grades closely match the NN modelling while showing appropriate smoothing.

In order to perform appropriate validations, a Nearest Neighbour (NN) model has been created in order to compare the de-clustered composites to the modelled grades. To validate the amount of smoothing in the model, the NN model is then corrected for block size by the indirect lognormal theoretical correction, based on the global variogram parameters and mean grades for each domain.

14.9.1 Global Grade Validation

Resource validation to ensure there is no global bias has been done by comparing NN grades to those of the final grade interpolation. The tables below summarize this comparison by Domain, illustrating that the difference in Au grades by domain is within 1% overall. The positive difference values of Au grade from Domains 6 and 7 are considered immaterial due to material within these domains being almost entirely (over 95%) below cutoff, with a mean grade of less than 0.08 g/t Au. For Ag, the comparison shows mean modelled grades within 4% for all domains and modelled grades virtually identical to the de-clustered composites overall.

Table 14-19 Summary of Au Grade Comparison with De-Clustered Composites by Domain

Model	Parameter	UARG-S	UARG-N	TUFF-S	TUFF-N	LARG	SLTST-S	SLTST-N	NZ-ARG	ALL
		DOM1	DOM2	DOM3	DOM4	DOM5	DOM6	DOM7	DOM8	
NN	Num Samples	21,067	12,012	41,733	10,121	146,144	6,391	23,441	36,518	297,427
	Min (gpt)	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001	0.001
	Max (gpt)	50.441	73.611	41.180	32.012	125.459	10.748	15.086	16.671	125.459
	Wtd. Mean (gpt)	0.409	0.300	0.220	0.231	0.185	0.059	0.068	0.232	0.206
	Weighted CV	2.886	4.871	3.942	4.509	7.151	6.476	6.250	2.087	5.404
2019 MODEL - MIK	Num Samples	21,067	12,012	41,733	10,121	146,144	6,391	23,441	36,518	297,427
	Min (gpt)	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
	Max (gpt)	5.196	6.291	8.255	4.742	14.310	2.800	3.118	2.871	14.310
	Wtd. Mean (gpt)	0.397	0.278	0.234	0.221	0.179	0.064	0.078	0.228	0.203
	Weighted CV	1.147	1.380	1.909	1.402	2.334	2.746	2.416	1.008	1.924
% Diff.	1-NN/MIK	-3.2%	-7.8%	6.0%	-4.7%	-3.0%	8.4%	12.3%	-1.8%	-1.2%

Table 14-20 Summary of Ag Grade Comparison with De-Clustered Composites by Domain

Model	Parameter	UARG-S DOM1	UARG-N DOM2	TUFF-S DOM3	TUFF-N DOM4	LARG DOM5	SLTST-S DOM6	SLTST-N DOM7	NZ-ARG DOM8	ALL
NN	Num Samples	21067	12012	41733	10121	146144	6391	23441	36518	297427
	Num Missing	0	0	0	0	0	0	0	0	0
	Min (gpt)	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
	Max (gpt)	53.440	24.800	29.700	19.800	13.390	13.920	13.390	35.240	53.440
	Wtd. Mean (gpt)	0.730	0.989	0.414	0.436	0.566	0.428	0.398	0.640	0.562
	Weighted CV	1.533	0.996	1.819	1.651	1.124	1.287	1.208	1.544	1.361
ID3	Num Samples	21067	12012	41733	10121	146144	6391	23441	36518	297427
	Num Missing	0	0	0	0	0	0	0	0	0
	Min (gpt)	0	0	0	0	0	0	0	0	0
	Max (gpt)	38.9	12.3	12	8.5	10.8	6.4	13	14.7	38.9
	Wtd. Mean (gpt)	0.73	1.03	0.41	0.44	0.57	0.43	0.41	0.64	0.56
	Weighted CV	1.11	0.75	1	0.95	0.78	0.84	0.74	0.98	0.92
Difference (%)	1-NN/ID3 2019	0.0%	4.0%	-1.0%	0.9%	0.7%	0.5%	2.9%	0.0%	-0.4%

14.9.2 Grade-Tonnage Curves

Grade-Tonnage curves have been created to compare the Au-MIK and Ag-ID3 interpolated grades with de-clustered composite grades. The de-clustered composites have been corrected for the Volume-Variance effect by applying the Indirect lognormal Correction (ILC) to the NN grades. This correction applies a factor to reduce the variance based on the block size (which is similar to the Selective Mining Unit or SMU) in order to ensure that the modelled grades have had appropriate smoothing applied. Figure 14-4 and Figure 14-5 illustrate this comparison for Au and Ag respectively showing increased smoothing (reduced grades and increased tonnage) compared to the uncorrected NN grade curves, but similar distribution compared to the theoretical NN-ILC grades.

14.9.3 Visual Comparisons

Further validation on local grade estimation has been done through visual comparisons of the modelled grades with the assays grade in section, plan and through three-dimensional checks, particularly of the higher grade zones. The figures below illustrate the block grades and assay grades in E-W cross-sections throughout the area of the Resource pit. Both the Resource pit and the reserve pit are illustrated on each section. Figure 14-6 through Figure 14-8 are sections for Au grade comparisons and Figure 14-9 through Figure 14-11 are the same sections comparing the Ag grades. Both Au and Ag grades appear to show similar grade distributions and values throughout the model.

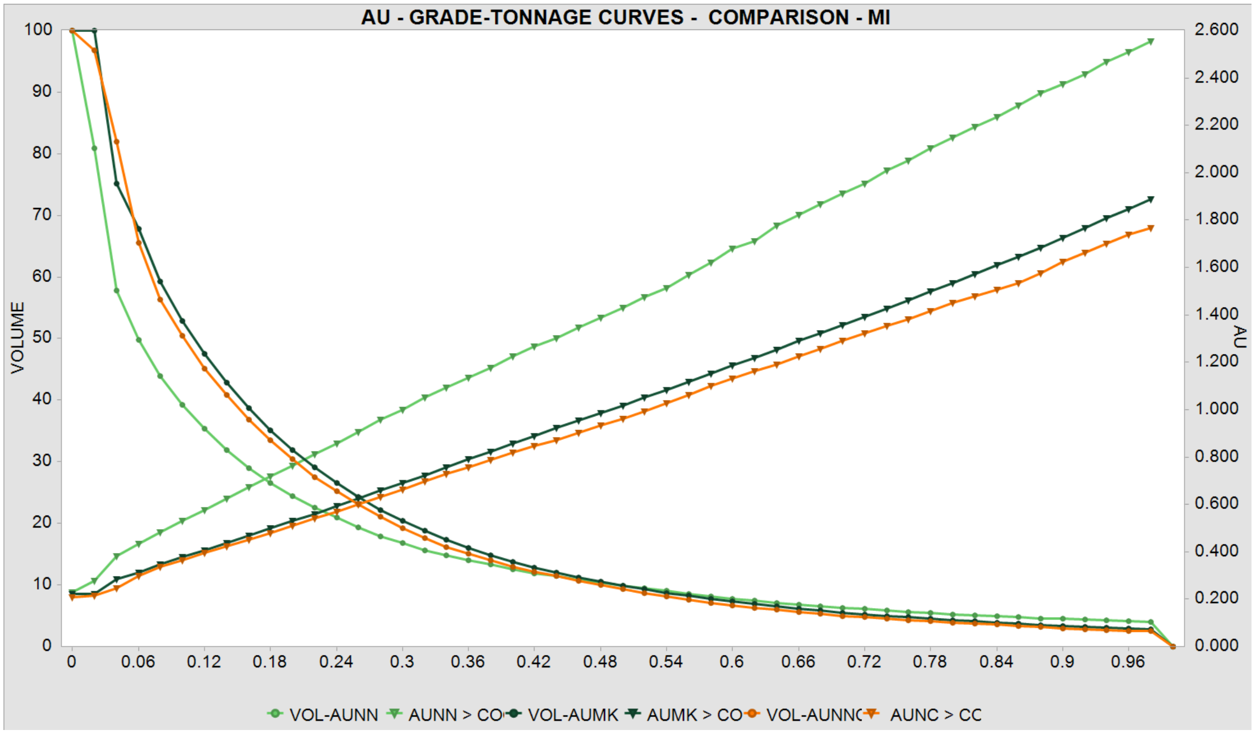


Figure 14-4 Grade-tonnage Curve compared to NN model and Theoretical NN-corrected – AU

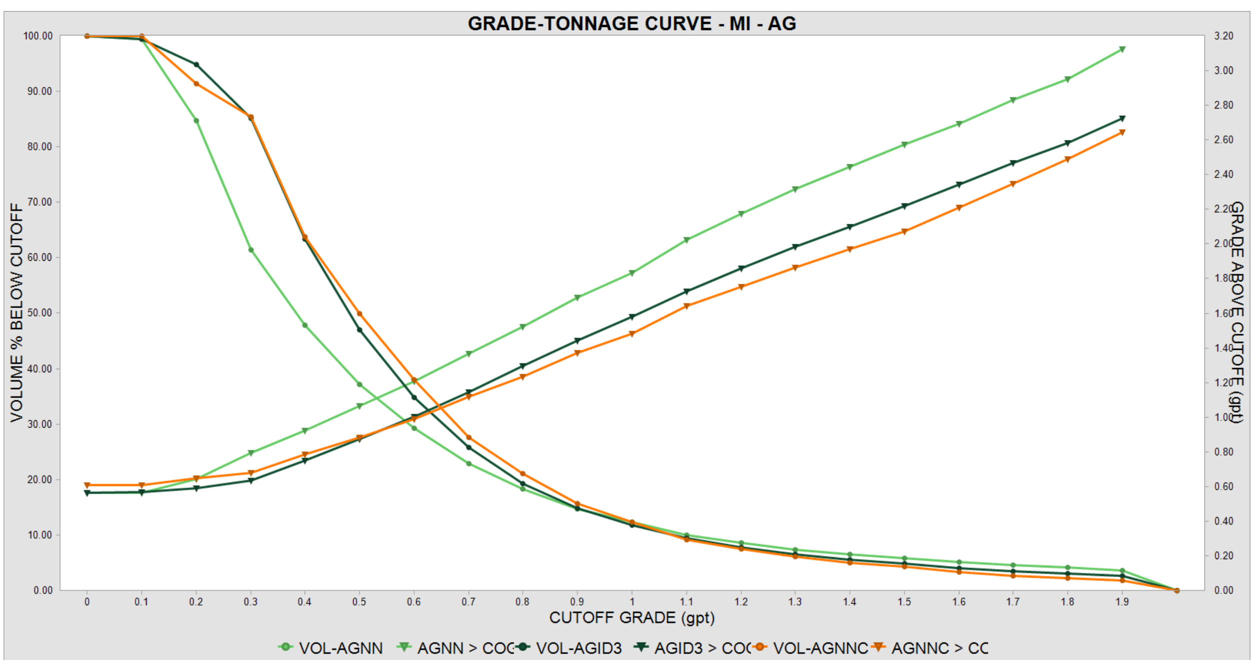


Figure 14-5 Grade-tonnage Curve compared to NN model and Theoretical NN-corrected – AG

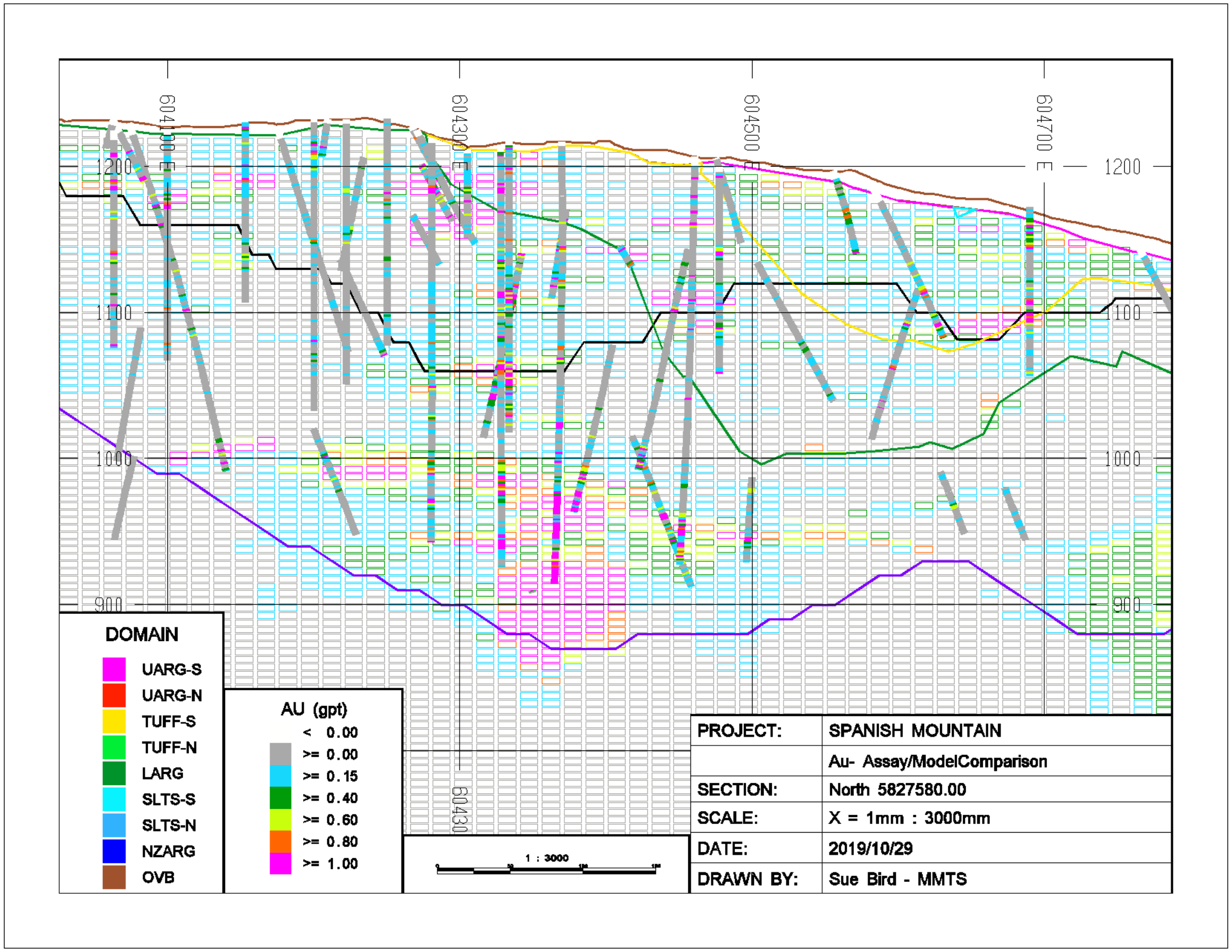


Figure 14-6 Comparison of Model Grades and Assay Grades – Au – Section 5827580N

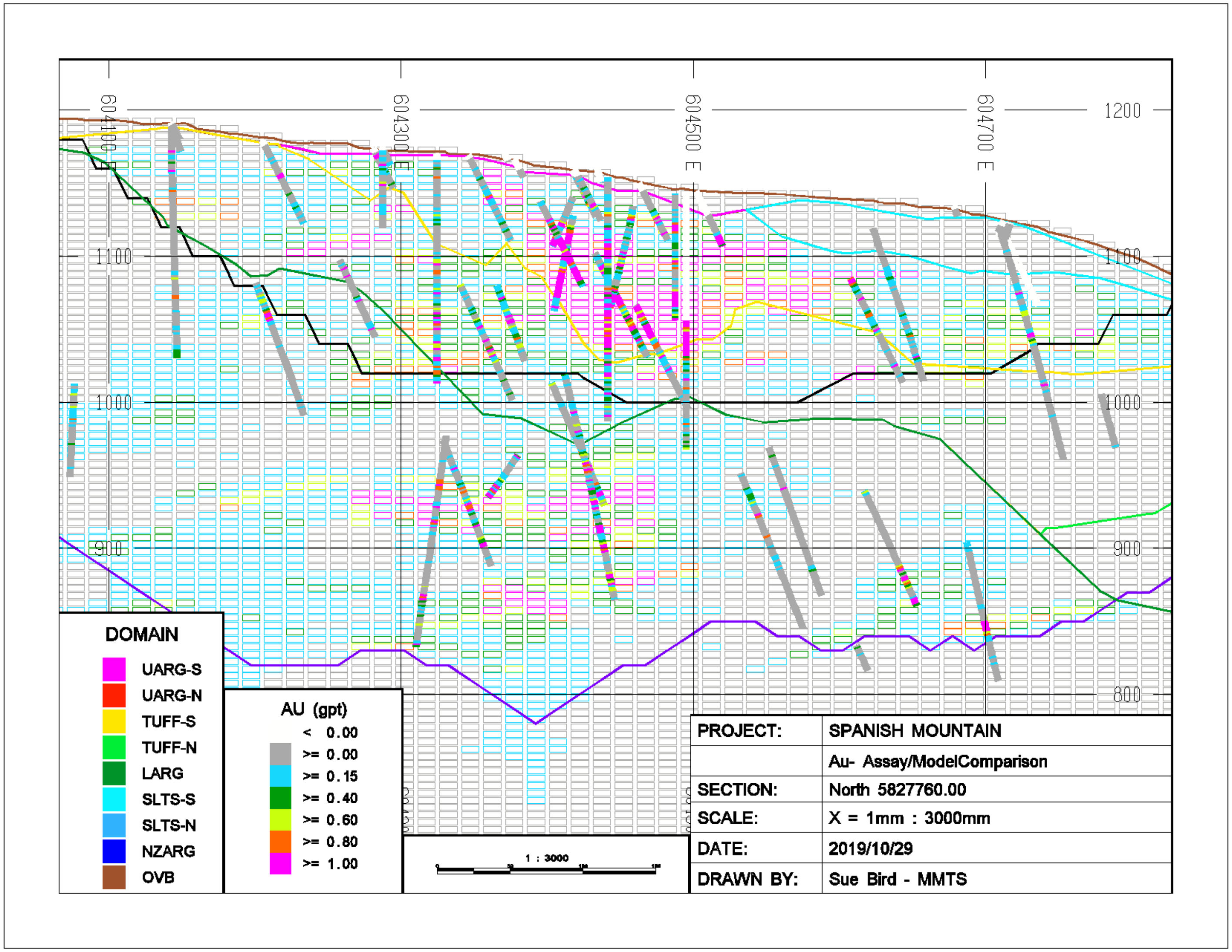


Figure 14-7 Comparison of Model Grades and Assay Grades – Au – Section 5827760N

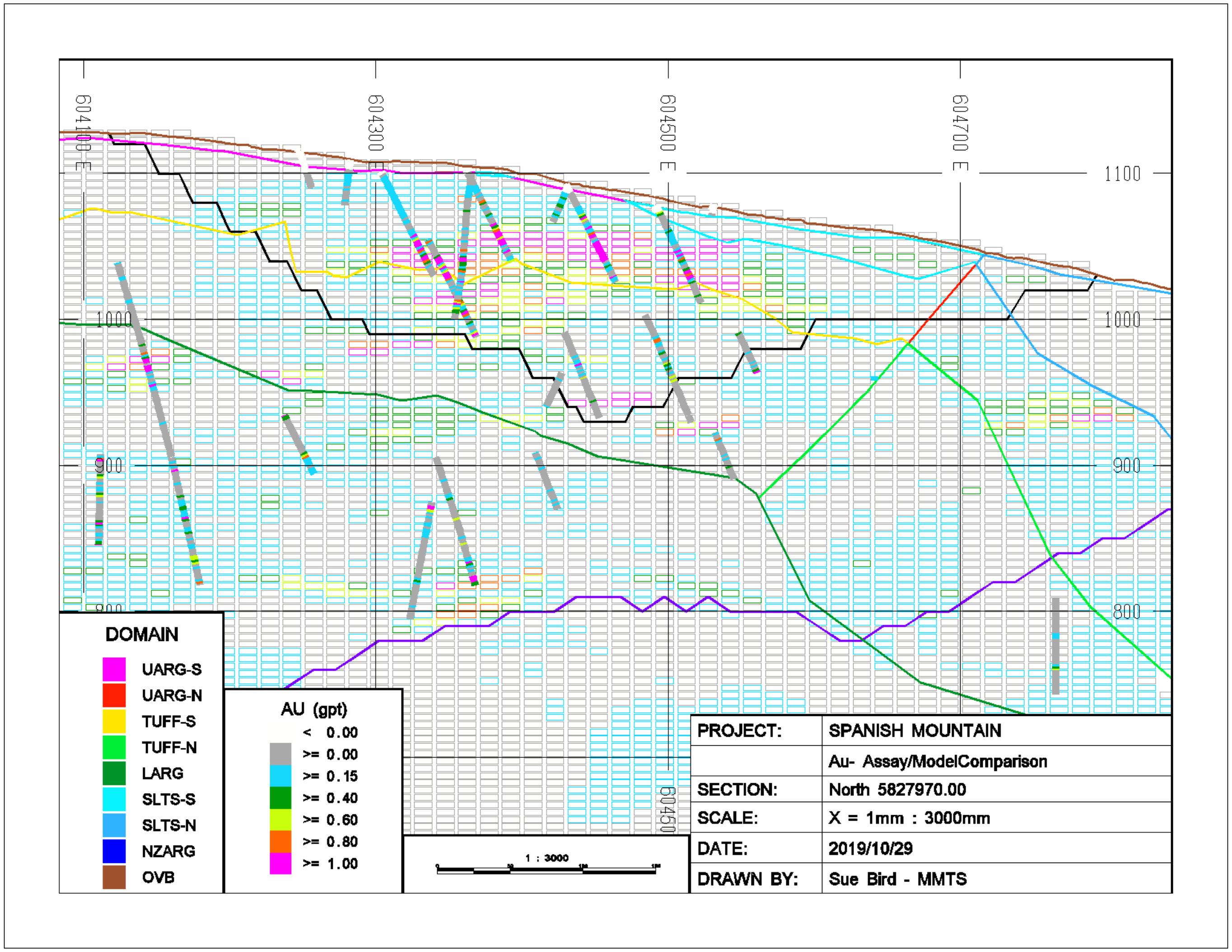


Figure 14-8 Comparison of Model Grades and Assay Grades – Au – Section 5827970N

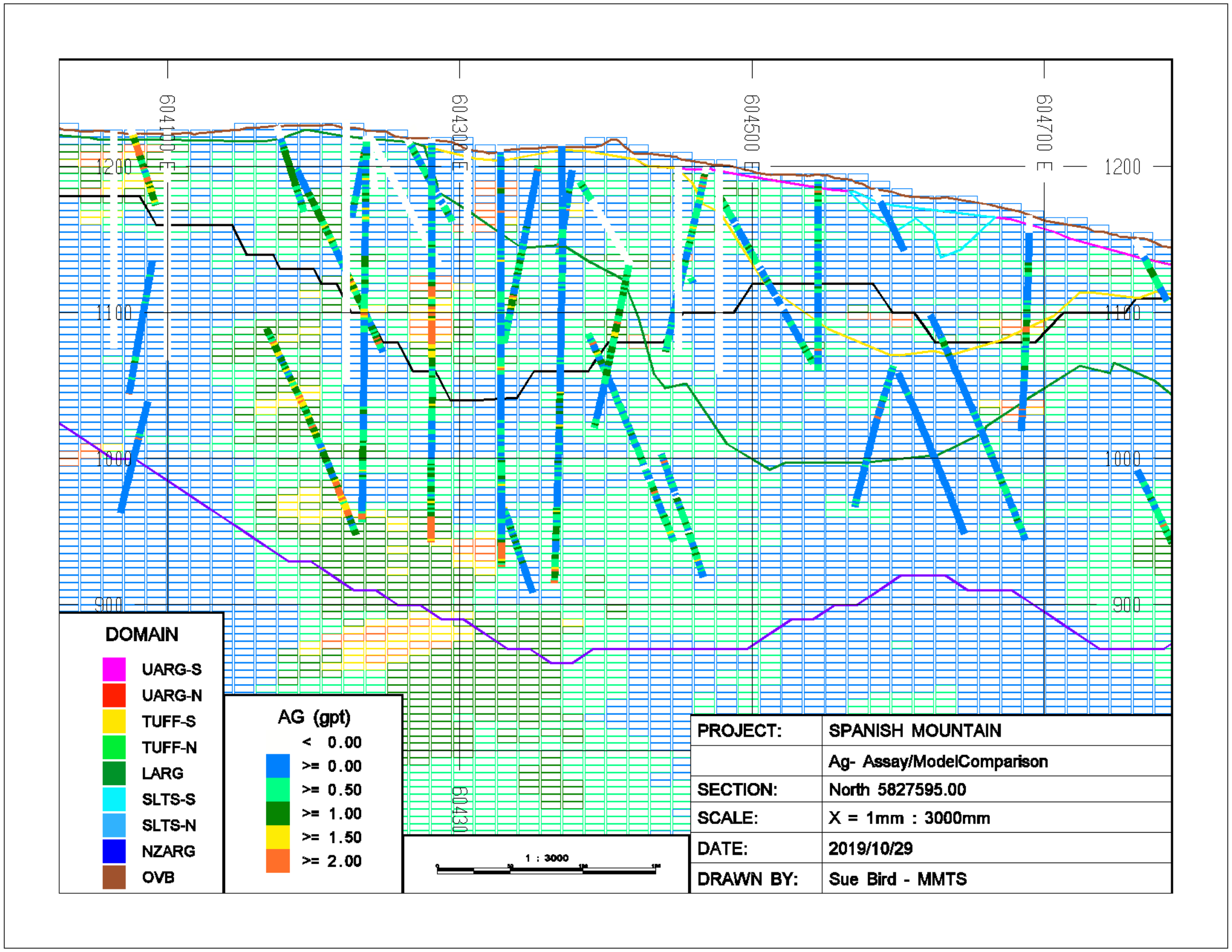


Figure 14-9 Comparison of Model Grades and Assay Grades – Ag – Section 5827580N

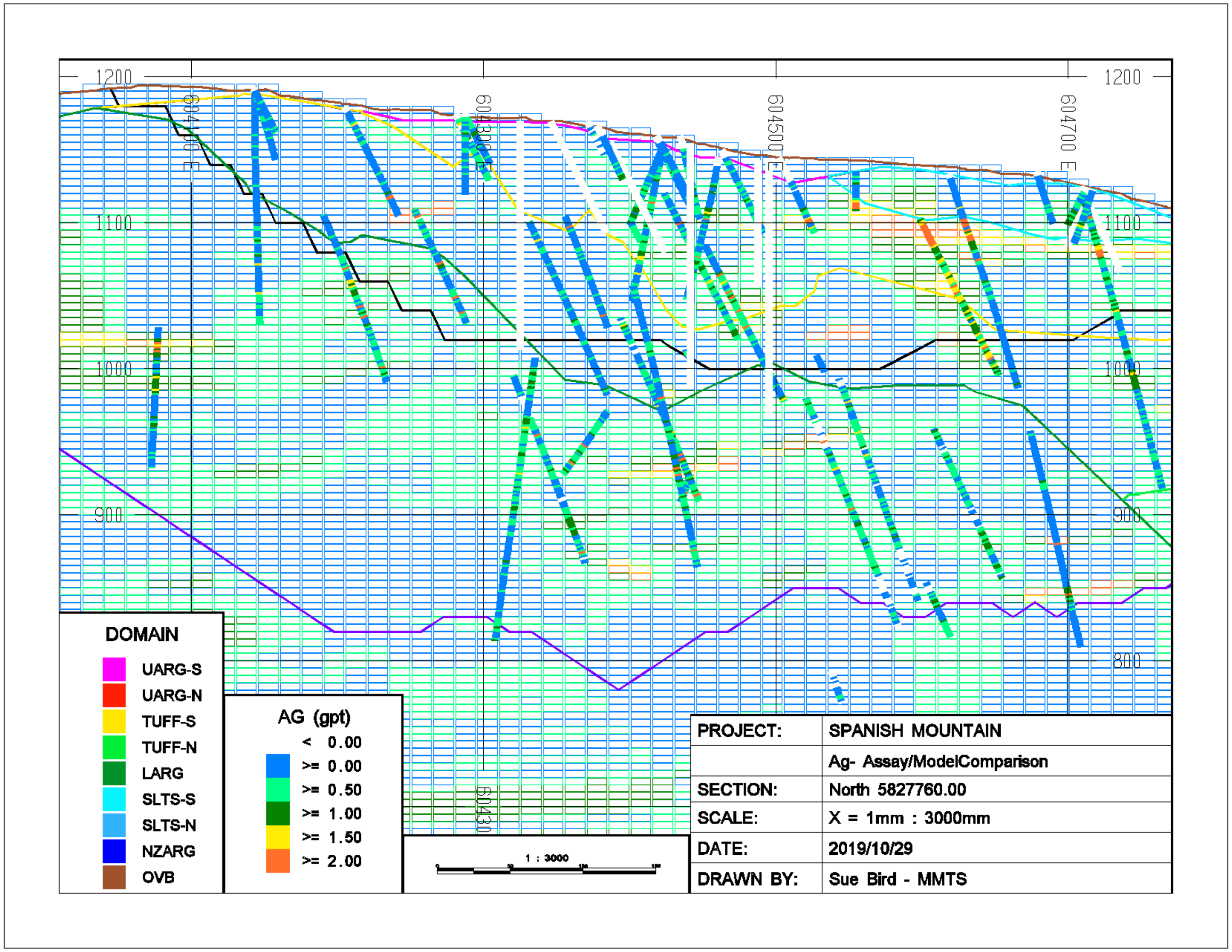


Figure 14-10 Comparison of Model Grades and Assay Grades – Ag – Section 5827760N

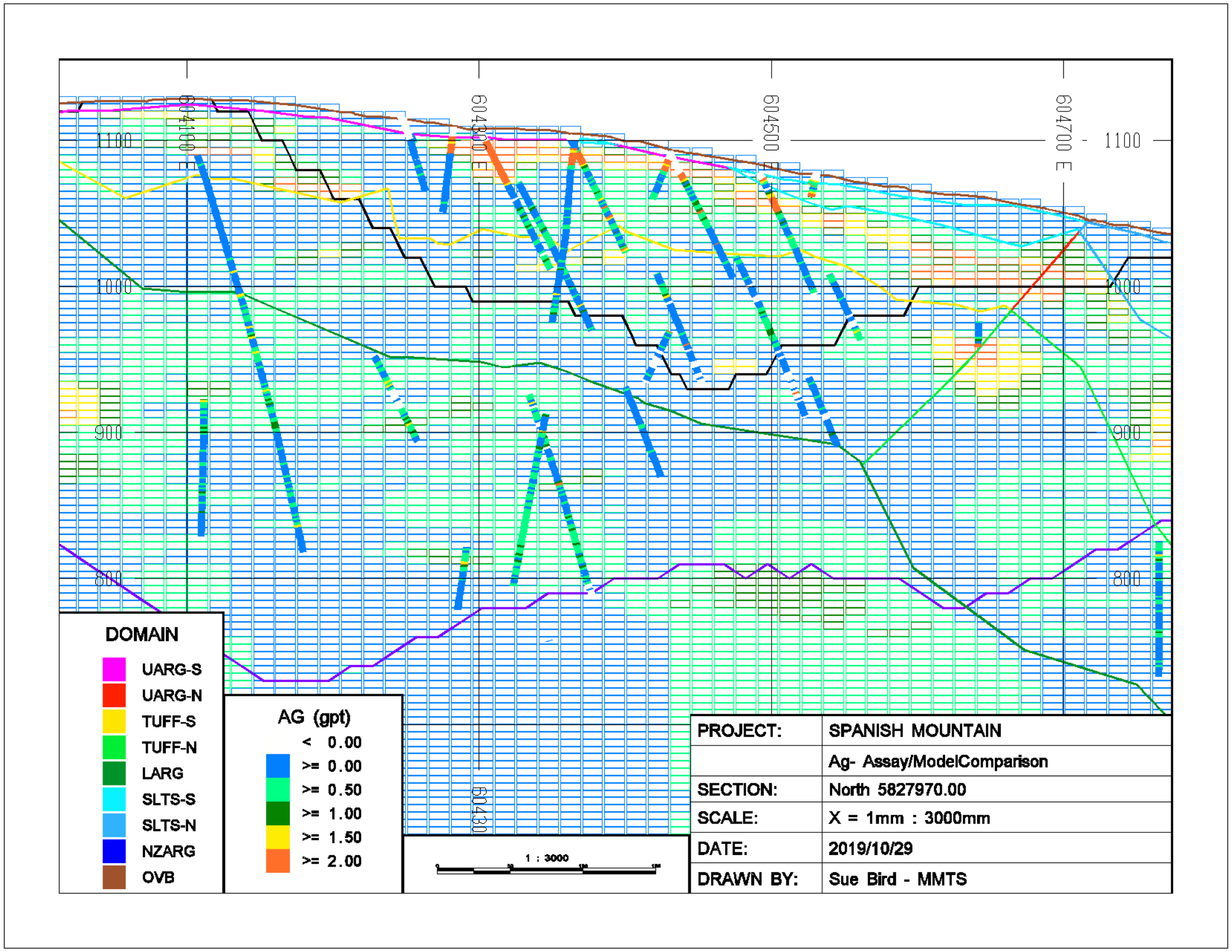


Figure 14-11 Comparison of Model Grades and Assay Grades – Ag – Section 5827970N

14.9.4 **Swath Plots**

Swath plots have been used to further examine the grade distribution throughout the model by comparing the de-clustered composites (NN model) grades with the Au-MIK and Ag-ID3 grades. Figure 14-12 through Figure 14-14 are Swath plots for Au, with Figure 14-15 through Figure 14-17 the swath plots for Ag. As can be seen in the plots, the Au-NN grades (de-clustered composites) are more erratic than the Au-MIK grades, indicating more smoothing for the modelled grades, but overall the mean grade trends follow each other in the three principal plotted directions. The Ag-NN grades are more similar to the AG-ID3 grades, which is to be expected from the lower Coefficients of Variation for Ag of the original drillhole data. All plots indicate that the interpolated grades used for the resource estimate follow the original data.

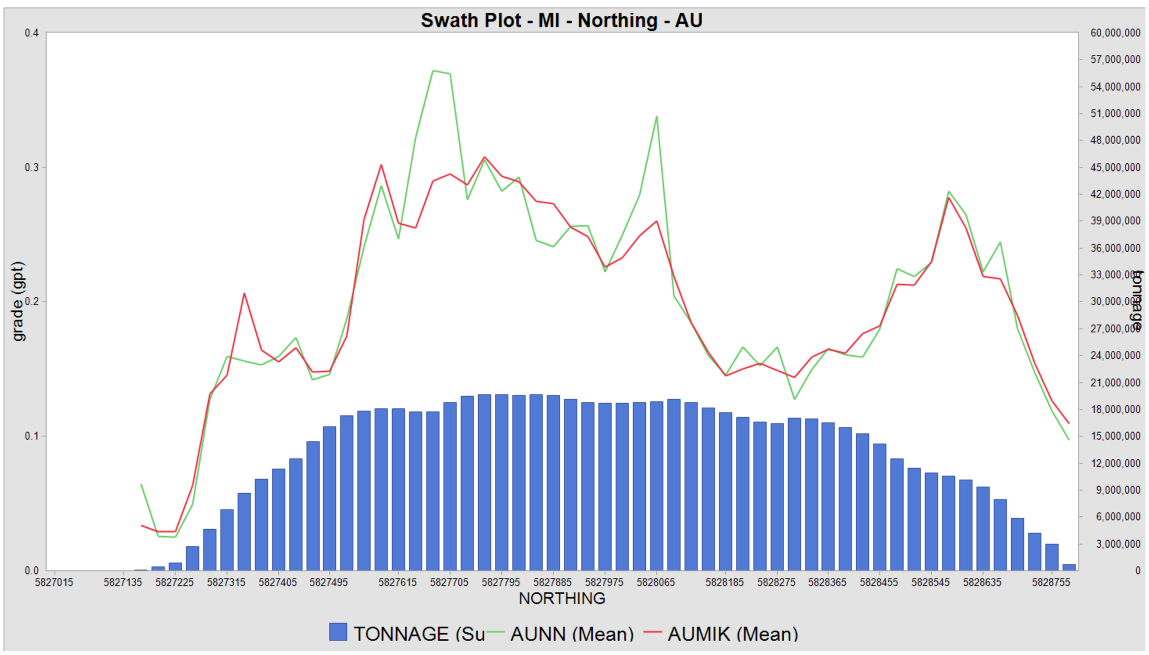


Figure 14-12 Swath Plot AU – Northing

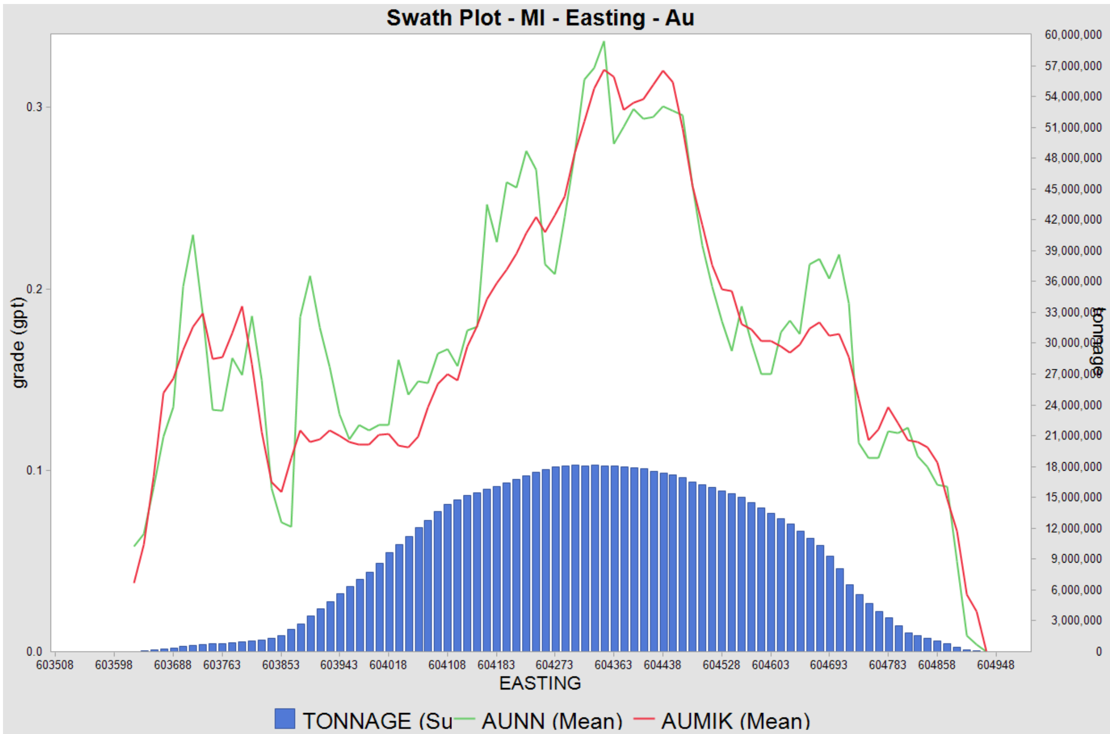


Figure 14-13 Swath Plot AU – Easting

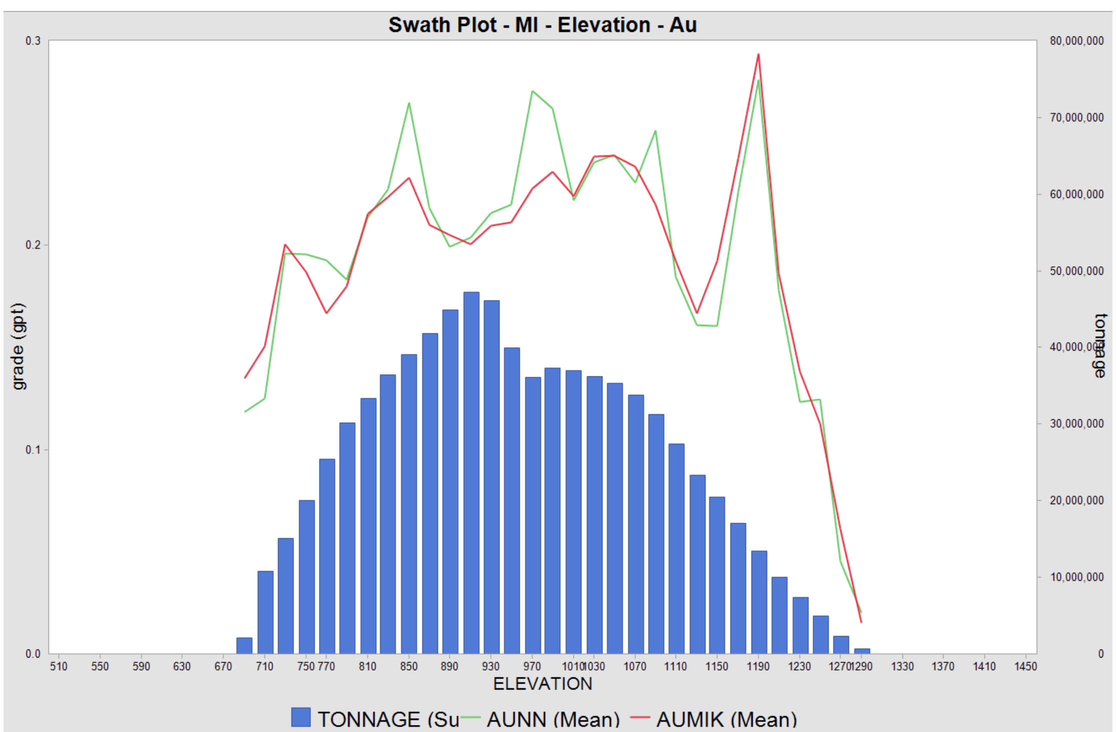


Figure 14-14 Swath Plot AU – Elevation

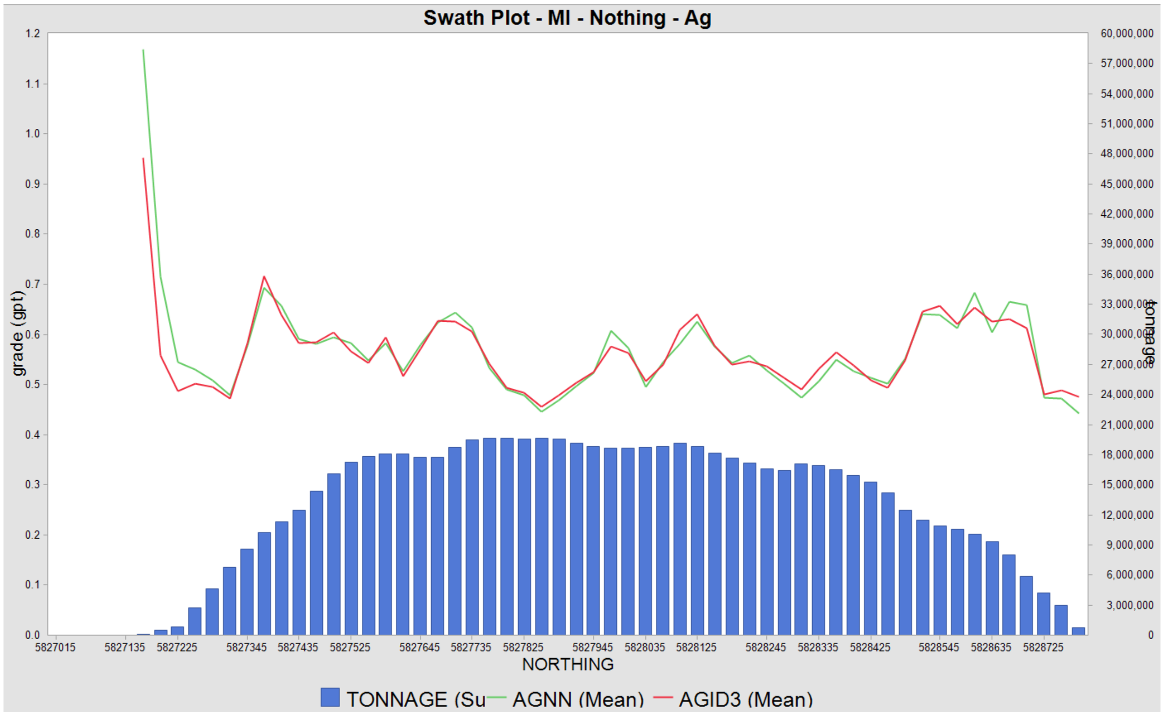


Figure 14-15 Swath Plot AG – Northing

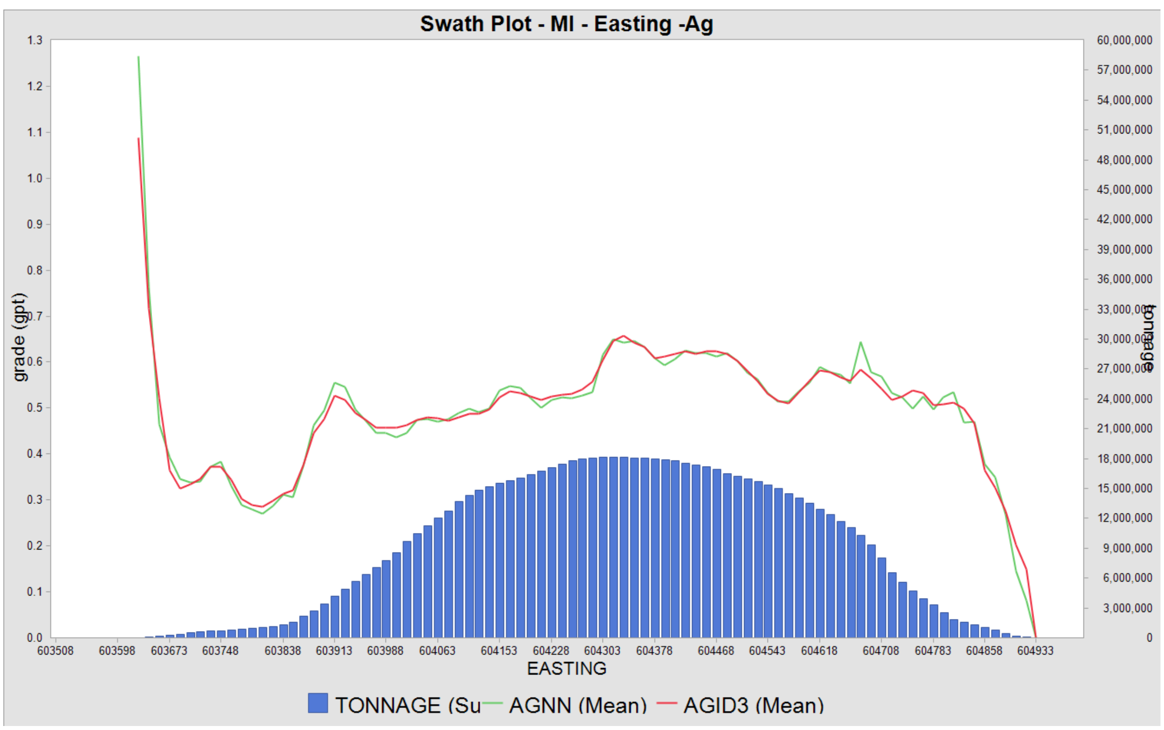


Figure 14-16 Swath Plot AG – Easting

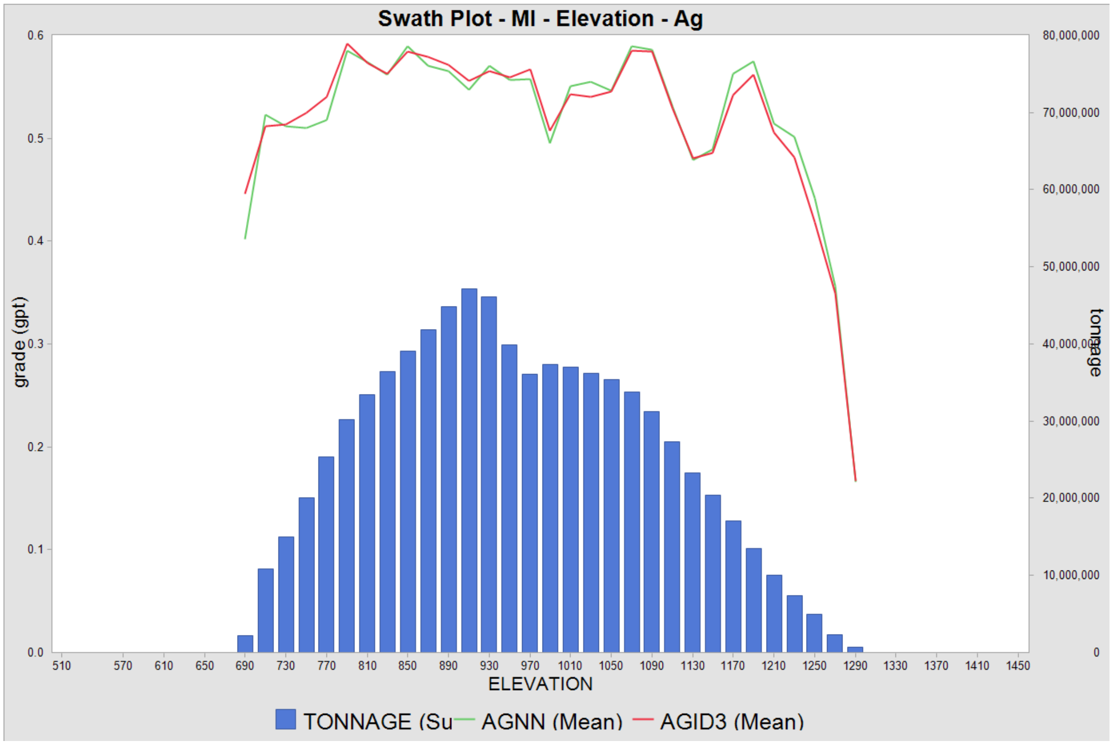


Figure 14-17 Swath Plot AG – Elevation

14.10 Classification

Classification is based on the variography, with the distances to at least two drillholes required to be within 15 m to be considered Measured, and to be within 35-60 m to be considered Indicated. This distance requirement is then followed by visual inspection of the preliminary classes in order to create cohesive Measured, Indicated and Inferred volumes of rock. The Inferred blocks are all other blocks that have been interpolated, except in areas below and outside the areal extent of the current drilling. These blocks have been removed from the Inferred category to limit extrapolation of data.

In reviewing of the QA/QC data for all years of drilling used in this resource estimate, a few potential issues with 2006 duplicate and standard data have been noted. Therefore, in order to remain somewhat conservative when evaluating the classification, the 2006 data were not used for Classification purposes.

Figure 14-18 illustrates the Classification through the central portion of the deposit also showing the drillhole data, pits and domain boundaries.

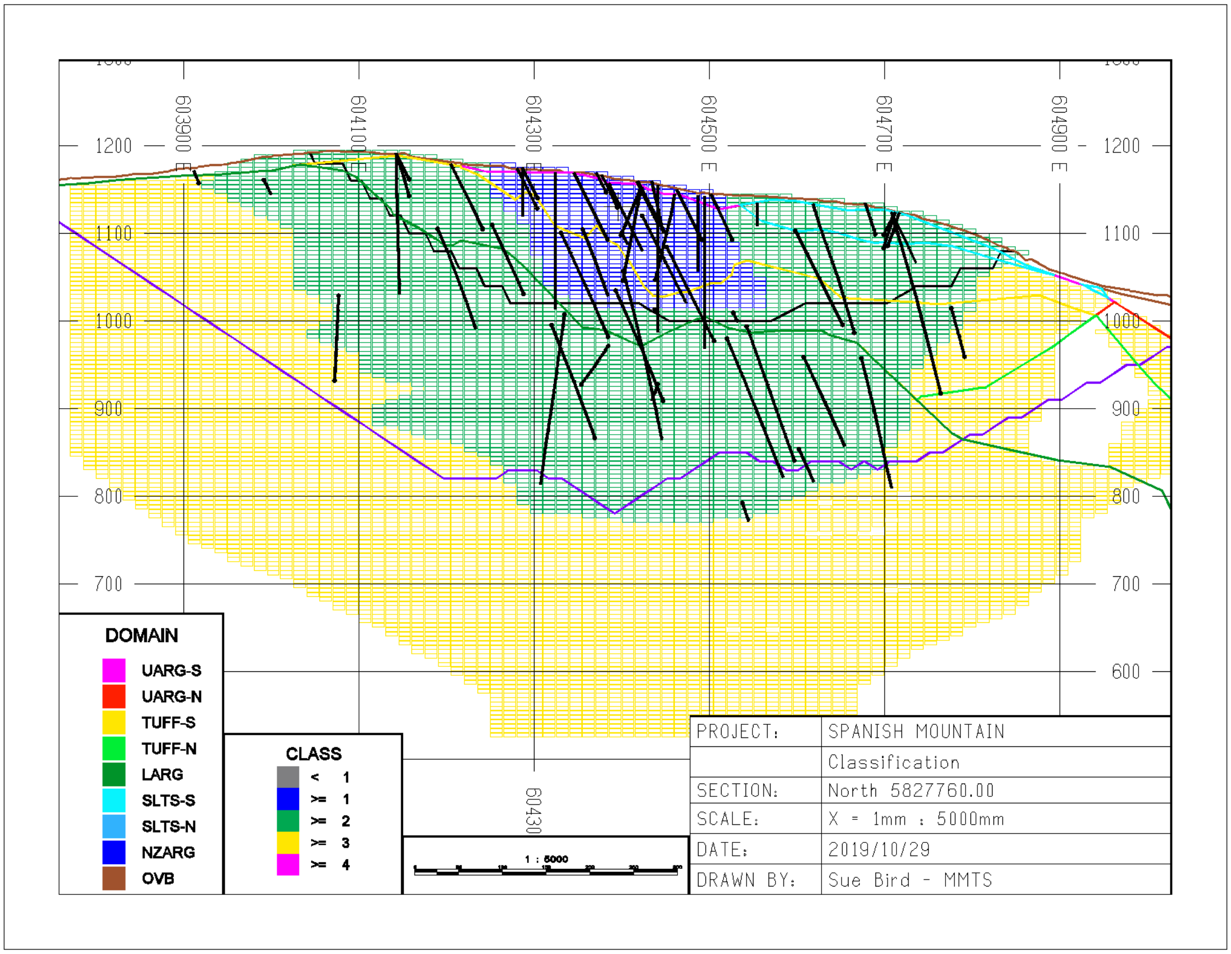


Figure 14-18 Classification and Drillhole Data +/- 15m from Section

14.11 Assessment of Reasonable Prospect of Eventual Economic Extraction Pit

The resource for the Spanish Mountain deposit has been confined within an open pit shape to define "reasonable prospects of eventual economic extraction" using the price, recovery and payable input parameters summarized below.

The Mineral Resource Estimate at a 0.15 g/t Au cut-off is based on conventional open pit mining with the following assumptions used to determine the cut-off grade:

- Gold Price = US$1,275/oz;
- Exchange Rate = 0.75 US$:1 C$; Payable Gold = 99.8%; Offsite Costs = US$4/oz; Royalties = 1.5%;
- Process Costs (including G&A costs) = $7.25 /t
- Process Recovery = 89%

The Lerchs-Grossman (LG) algorithm is applied to the block model in order to define the open pit shape. Along with the cut-off grade assumptions listed above, the LG algorithm utilizes a $2.20/t mining cost and overall slope angles ranging from 20° to 43°.

It should be noted that the entire block model is within SMG claim boundaries.

Figure 14-19 illustrates the "reasonable prospects of eventual economic extraction" pit shape and the drillholes used in the resource estimate.

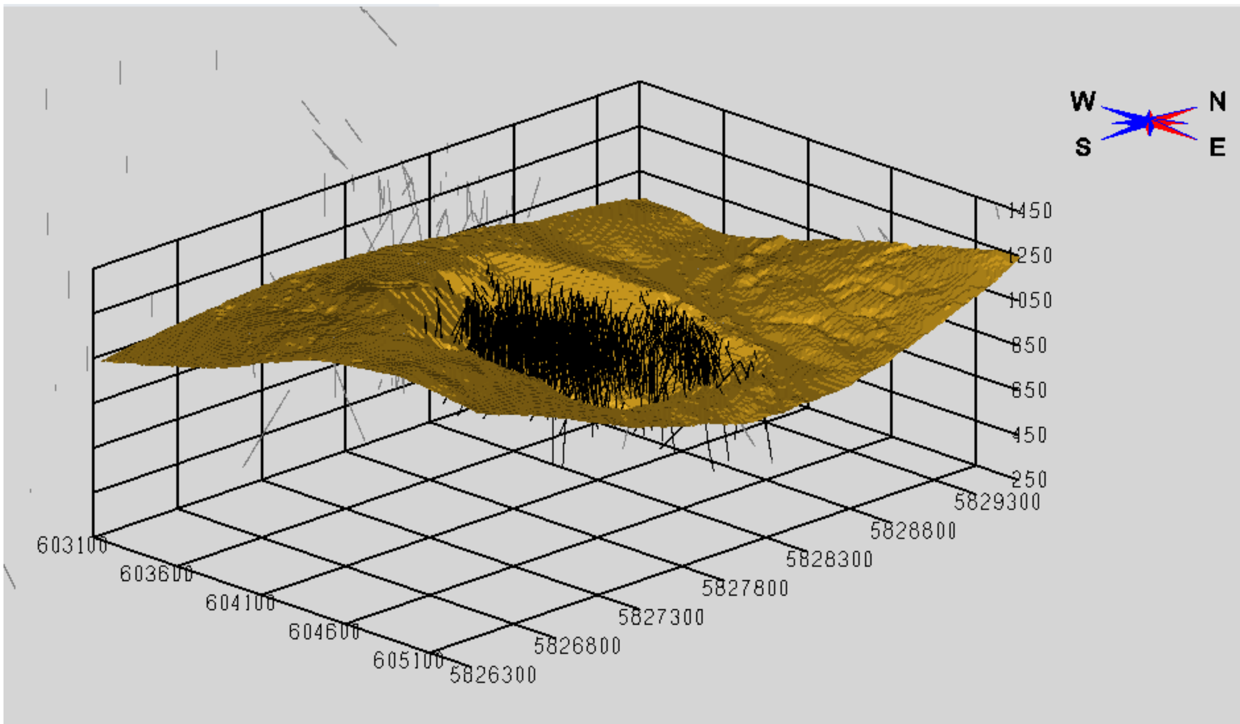


Figure 14-19 3D View Looking NW of the "Reasonable Prospects" Pit

15.0 Mineral Reserve Estimates

This Section is not relevant to the Technical Report.

16.0 Mining Method

The Spanish Mountain mine operations are planned to a Scoping level of accuracy using conventional open pit mining methods. The following section describes the mine design and mine engineering for the project, including pit optimization, open pit phasing and design, ore and waste stockpile design, annual mine production plans and a simple description of the planned open pit operations.

16.1 Summary

The open pit is designed for approximately thirteen years of operation. The subset of Mineral Resources contained within the designed open pit, summarized in Table 16-1 with a 0.40 g/t gold cut-off, forms the basis of the mine plan and production schedule. All Inferred Resource Class material is treated as waste, and mill feed is made up completely of Measured and Indicated Resource Class materials. There is no certainty that the economic results from this PEA will be realized.

Table 16-1 Mining ROM Production

	Unit	Amount
Mill Feed	kt	39,097
Gold Grade	g/t	1.00
Silver Grade	g/t	0.74
Waste Material	kt	138,541
Strip Ratio	t/t	3.5

- *The PEA Mine Plan and Mill Feed estimates are a subset of the October 10, 2019 Mineral Resource estimates (described in Table 14-1 and* Table 14-2*) and are based on open pit mine engineering and technical information developed at a Scoping level for the Spanish Mountain Gold deposit.*
- *Mill Feed estimates are mined tonnes and grade, the reference point is the primary crusher.*
- *Mining recovery of 97% and external mining dilution of 10.9% at 0.34 g/t Au grade is applied in addition to the modelled in-block dilution.*
- *Factors that may affect the estimates include: metal price assumptions, changes in interpretations of mineralization geometry and continuity of mineralization zones, changes to kriging assumptions, metallurgical recovery assumptions, operating cost assumptions, confidence in the modifying factors, including assumptions that surface rights to allow mining infrastructure to be constructed will be forthcoming, delays or other issues in reaching agreements with local or regulatory authorities and stakeholders, and changes in land tenure requirements or in permitting requirement.*
- *Estimates have been rounded and may result in summation differences.*

The crusher will be fed with material from the pit and supplemented by the ROM stockpile in Year 11, at an average rate of 10,000 t/d.

Figure 16-1 shows a plan view of the preliminary 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 five 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.

An elevated 0.45 g/t cut-off grade is employed to enhance the economics of the project. 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 either reclaimed at the end of the mine life (in Year 11), or blended with the run-of-mine (ROM) feed if an appropriate opportunity arises. Any stockpiled mineralized material not reclaimed to the crusher in this PEA mine plan is considered waste in the total tonnages shown in Table 16-1.

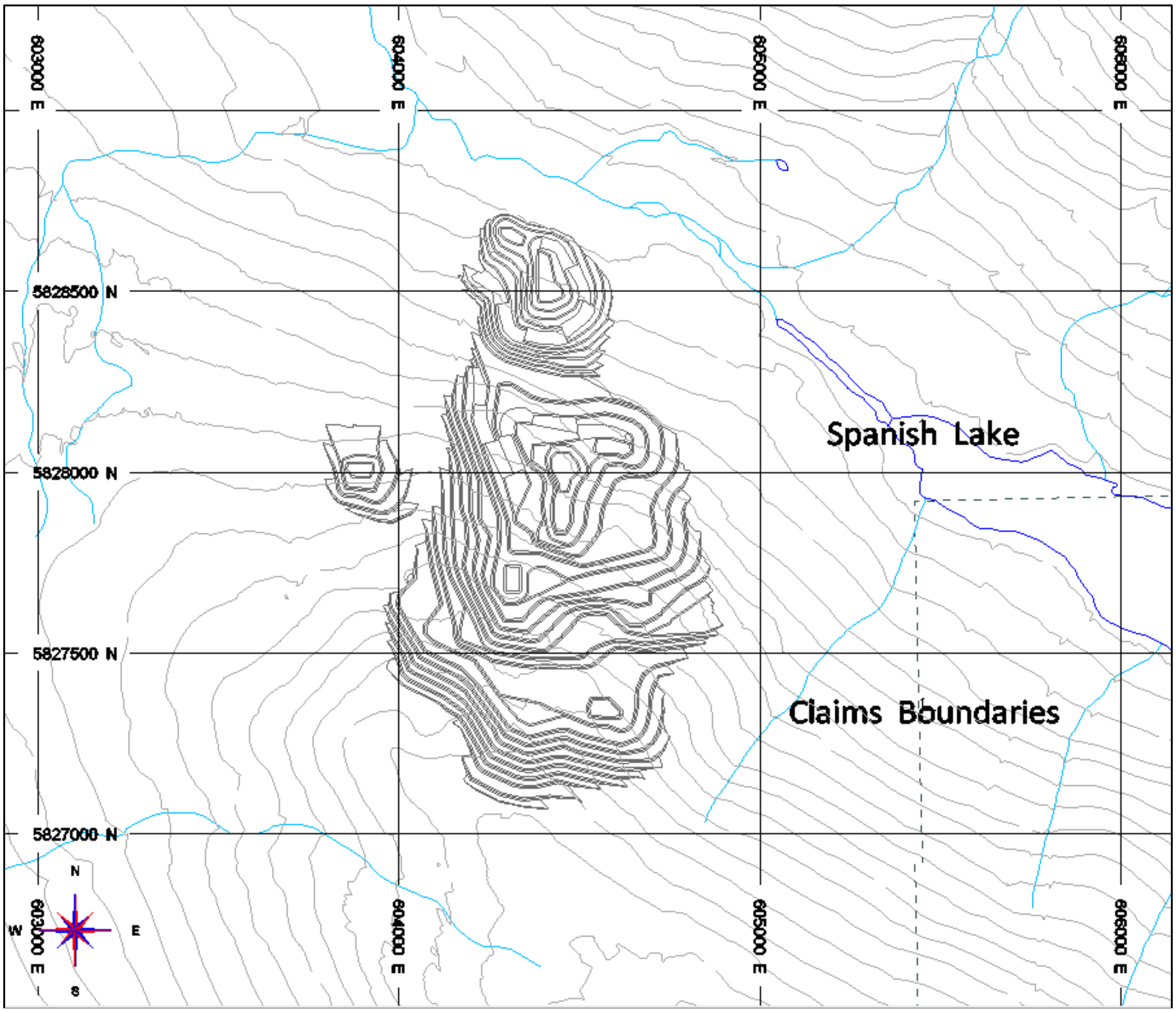


Figure 16-1 Ultimate Pit Design - Plan View

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 of in the tailings pond. A small amount of the PAG waste rock will be used for upstream construction of the dam embankment where it will be eventually submersed. Suitable NPAG pit waste rock will also be hauled to the TSF for dam construction, as needed.

Figure 1-3 shows the mine layout for the pit, WRSF's, and ROM stockpile.

16.2 Mine Planning 3D Block Model

Mine planning work is based on the resource 3DBM as described in Section 14. Extra items are added to the resource 3DBM to carry out open pit mine planning.

16.2.1 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 contract edges that are identified in each block at a 0.40 g/t gold cut-off grade. For each edge, a diluting wedge is assumed that carries a gold grade of 0.34 g/t and 0.61 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 the overall in pit resource out by an additional 10.9%.

Table 16-2 Dilution % based on number of waste/ore contacts in block

Number of waste / ore contact edges (using 0.40 g/t gold cut-off)	Dilution % applied to block
1	9%
2	18%
3	28%
4	39%

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.

16.2.2 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.

16.3 Open Pit Optimization Method

Economic pit limits for this study of the Spanish Mountain deposit are determined using a Lerchs-Grossman (LG) evaluation.

The economic pit limit is selected after evaluating various LG 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 Net Smelter Price

Net Smelter Price (NSP) is used in place of the Market Price for gold when running the LG optimizations, to consider all offsite costs to the project.

Using a gold market price of US$1,275/oz results in an NSP value of C$1,667/oz or C$53.60/g. Silver grades and associated net smelter value are not included in the LG analysis. The NSP calculation uses the inputs shown in Table 16-3:

Table 16-3 NSP Calculation Inputs

Description	Values	Units
Gold Price	$1,275	US$/oz
US Exchange rate	0.75	US$/C$
Payable Au	99.8%	%
Au Offsite Costs (Refining and Transport)	$4.00	US$/oz
Royalty	1.5%	

16.3.2 Process Recovery

The process recovery assumptions are 91% for both the pit optimization and cut-off grade estimation.

16.3.3 LG Pit Operating Costs

Potential block revenues are calculated based on the NSP, process recovery, gold grade and mineralized percentage within each block.

Operating costs are used in conjunction with these potential block revenues to run the LG algorithm and generate open pit shells. The following operating costs are used in the LG analysis:

Table 16-4 LG Operating Cost Inputs

Operation	Cost
Base Ore Mining Cost (Pit Rim)	$2.15/t
Base Waste Mining Cost (Pit Rim)	$2.15/t
Incremental Haulage Cost	$0.015 per 5 m bench below 950 m model elev.
Processing Cost	$5.00/t
General/Administration Cost	$2.25/t

16.3.4 **Pit Slope Angles**

The pit slopes are designed based on preliminary recommendations developed from geotechnical drilling carried out in 2010 and 2011 (BGC 2012). 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. Table 16-5 and Figure 16-2 summarizes the relevant information from the BGC report, specifically the wall design criteria for each sector. In-pit ramps (29 m wide) and geotechnical berms (minimum 26 m wide) are included in the design where necessary to reduce the overall slope angles and facilitate geotechnical instrumentation and dewatering.

Groundwater pressures will have a significant effect on the stability of the pit slopes. Preliminary hydrogeological studies carried out (BGC 2012) indicate that significant dewatering efforts will be required to depressurize the open pit slopes. 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 LG pit shells conform to overall pit slope recommendations provided in Figure 16-2.

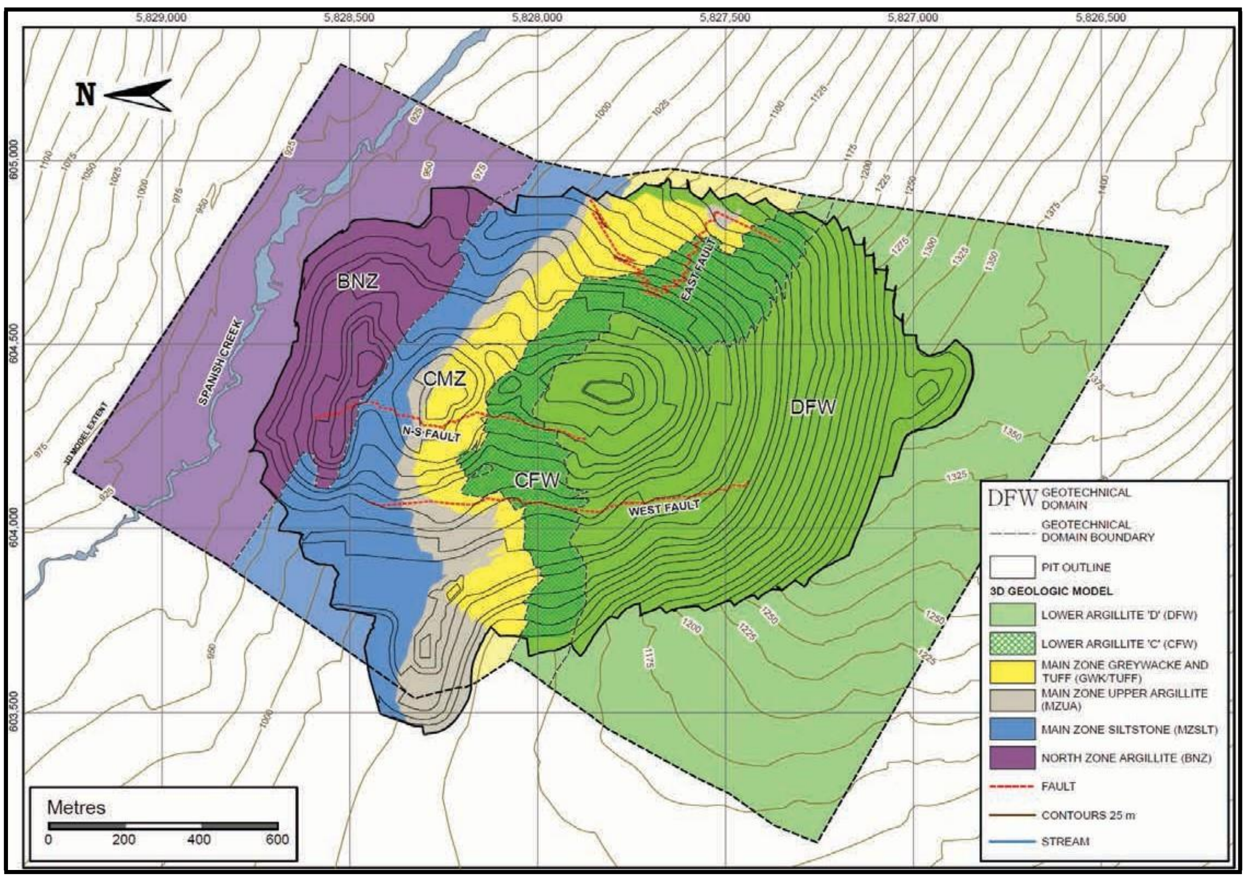


Figure 16-2 **Structural Domains for Pit Slope Designs**

Table 16-5 Pit Slope Design Recommendations

Domain	Design Sector	Azimuth Start (°)	Azimuth End (°)	Bench Height (m)	Bench Face Angle (°)	Berm Width (m)	Geotechnical Berm Maximum Spacing (m)	Inter Berm Angle (°)	Design Control	LG Input Overall Angle (°)
BNZ	BNZ-180	155	205	10	65	18.6	100	23	FB1-FC1	20
BNZ	BNZ-220	205	235	10	65	15.7	100	26	P-FC1	22
BNZ	BNZ-300	235	5	10	65	9	100	36	Bench geometry	31
BNZ	BNZ-020	5	35	10	65	10.7	100	33	P-FA2	28
BNZ	BNZ-046	35	57	10	65	15.7	100	26	FB2-FA2	23
BNZ	BNZ-089	57	120	10	65	17.6	100	24	FB2-FA2	21
BNZ	BNZ-130	120	140	10	65	14.1	100	28	P-FB2, FB1-FC1	25
BNZ	BNZ-148	140	155	10	65	15.7	100	26	FB1-FC1	22
CMZ	CMZ-2183	160	275	20	65	9.9	160	46	T-FA3	38
CMZ	CMZ-293	275	310	20	65	27.4	160	29	P-BFD1	25
CMZ	CMZ-339	310	7	20	65	34.2	160	25	FA3-BFD1	23
CMZ	CMZ-031	7	55	20	65	28.9	160	28	P-BFA1	25
CMZ	CMZ-090	55	125	20	65	34.2	160	25	BFA1-FB2	23
CMZ	CMZ-133	125	140	20	65	28.9	160	28	BFA1-FB2	25
CMZ	CMZ-150	140	160	20	65	20.8	160	34	BFA1-FB2	30
CFW	CFW-183	135	230	20	65	23.2	-	32	Multiple wedges	29
CFW	CFW-268	230	305	20	65	36.2	160	24	P-FC1	21
CFW	CFW-320	305	335	20	65	25.9	-	30	BFA1-FD1	27
CFW	CFW-005	335	35	20	65	34.2	-	25	BFA1-FD1	23
CFW	CFW-055	35	75	20	65	28.9	-	28	P-BFA1	25
CFW	CFW-088	75	105	20	65	13	160	42	Rockmass stability	37
CFW	CFW-120	105	135	20	65	16.6	-	38	P-FB2	33
DFW	DFW-188	155	220	20	65	13.9	160	41	Multiple wedges	37
DFW	DFW-2353	220	250	20	65	9.5	160	47	Bench geometry	43
DFW	DFW-261	250	272	20	65	12.2	160	43	P-FD1	39
DFW	DFW-284	272	295	20	65	19.1	160	35	FD2-FC1	32
DFW	DFW-320	295	345	20	65	23.2	160	32	FD2-FC1	30
DFW	DFW-025	345	65	20	65	36.2	160	24	FA1-FD2	23
DFW	DFW-103	65	140	20	65	28.9	160	28	FA1-FB2	26
DFW	DFW-148	140	155	20	65	20.8	160	34	FA1-FB2	31
Overb.		0	360	10	65	-	-	21		21

16.3.5 **Spanish Creek Restriction**

A mining restriction will limit mining activity south of Spanish Creek. The north end of the pit will be offset to avoid impacting the flow of the creek. The pit crest will be limited to the 915 masl on the south side of the creek. By applying these design criteria, the offset distance will generally be over 100 m from the creek; further hydrology and environmental work is required to confirm the adequacy of this offset distance.

16.4 Cut-off Gold Grade

The cut-off grade is chosen as the gold grade required to pay for processing costs and general and administration costs. The sum of these operating costs is estimated to be $6.07/t. Based on the NSP and process recovery formulas above; the economic gold cut-off grade is 0.15 g/t.
In order to boost mill feed grades an elevated gold cut-off grade of 0.45 g/t is also applied. Material between the economic cut-off grade and this elevated cut-off grade is stockpiled. Only a small portion of this stockpiled material is planned to be reclaimed back to the mill for processing at the end of the mine life. An opportunity exists to economically process this material in the future.

16.5 LG 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$200/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 LG Price Case pit shells using a 0.40 g/t cut-off gold grade. Various inflection points can be seen in the curve drawn in Figure 16-3 of cumulative inventory by pit case.

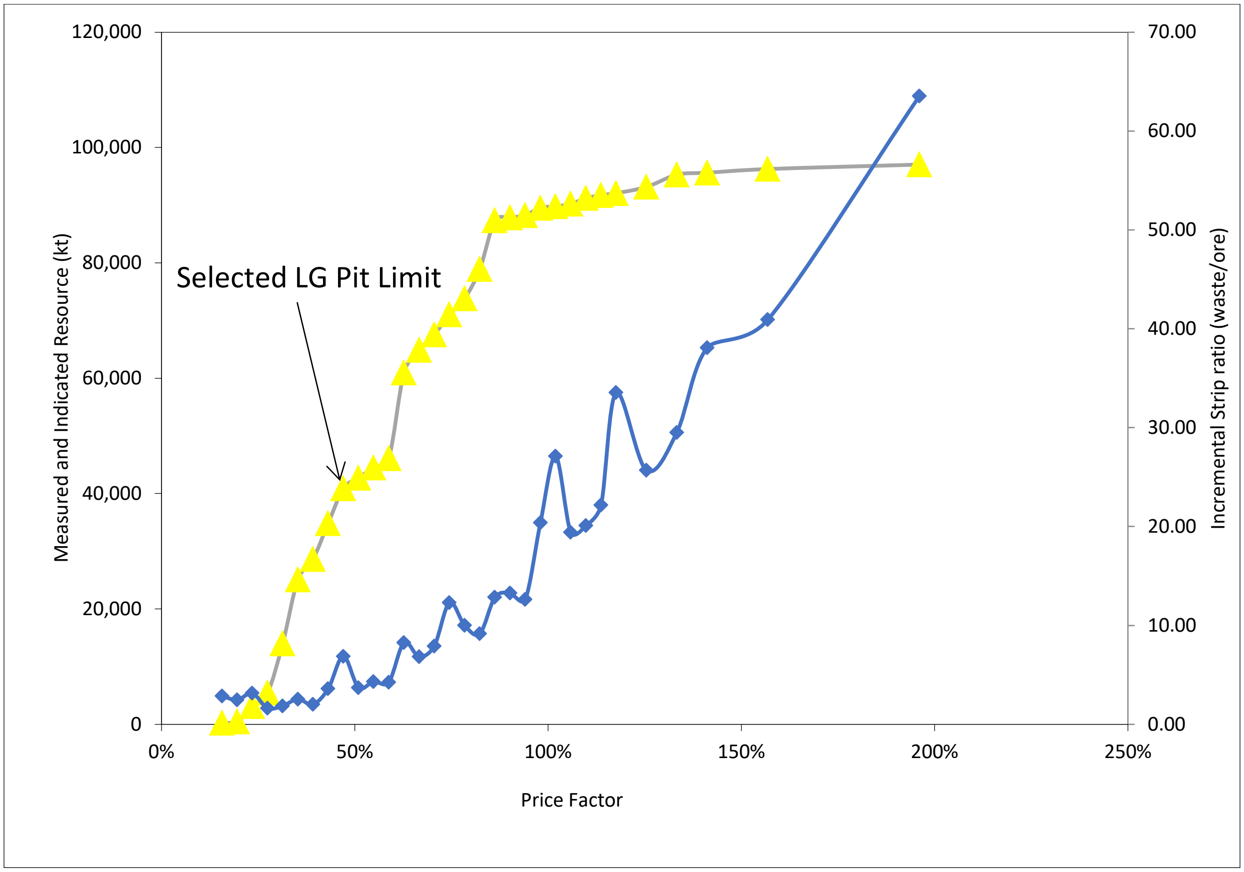


Figure 16-3 LG Price Cases Cumulative Inventory

16.5.1 **Selected Ultimate Pit Limits**

The pit shell generated at the 47% case is selected as the ultimate pit limit and is used for subsequent mine planning in this study. The subset of the Mineral Resources contained within this selected LG pit limit are shown in the Table below with a cut-off gold grade of 0.40 g/t. This LG shell target is used for further mine planning at Spanish Mountain Gold.

Table 16-6 LG Pit Delineated Contents

Input Gold Price	$450	US$/oz
Measured and Indicated Resource	40,862	kt
Gold Grade	1.00	g/t
Waste and Inferred Resource	128,332	kt
Strip Ratio	3.41	Waste / Resource
Total Pit Contents	169,194	kt

The following figures show plan and section views of this chosen ultimate pit shell.

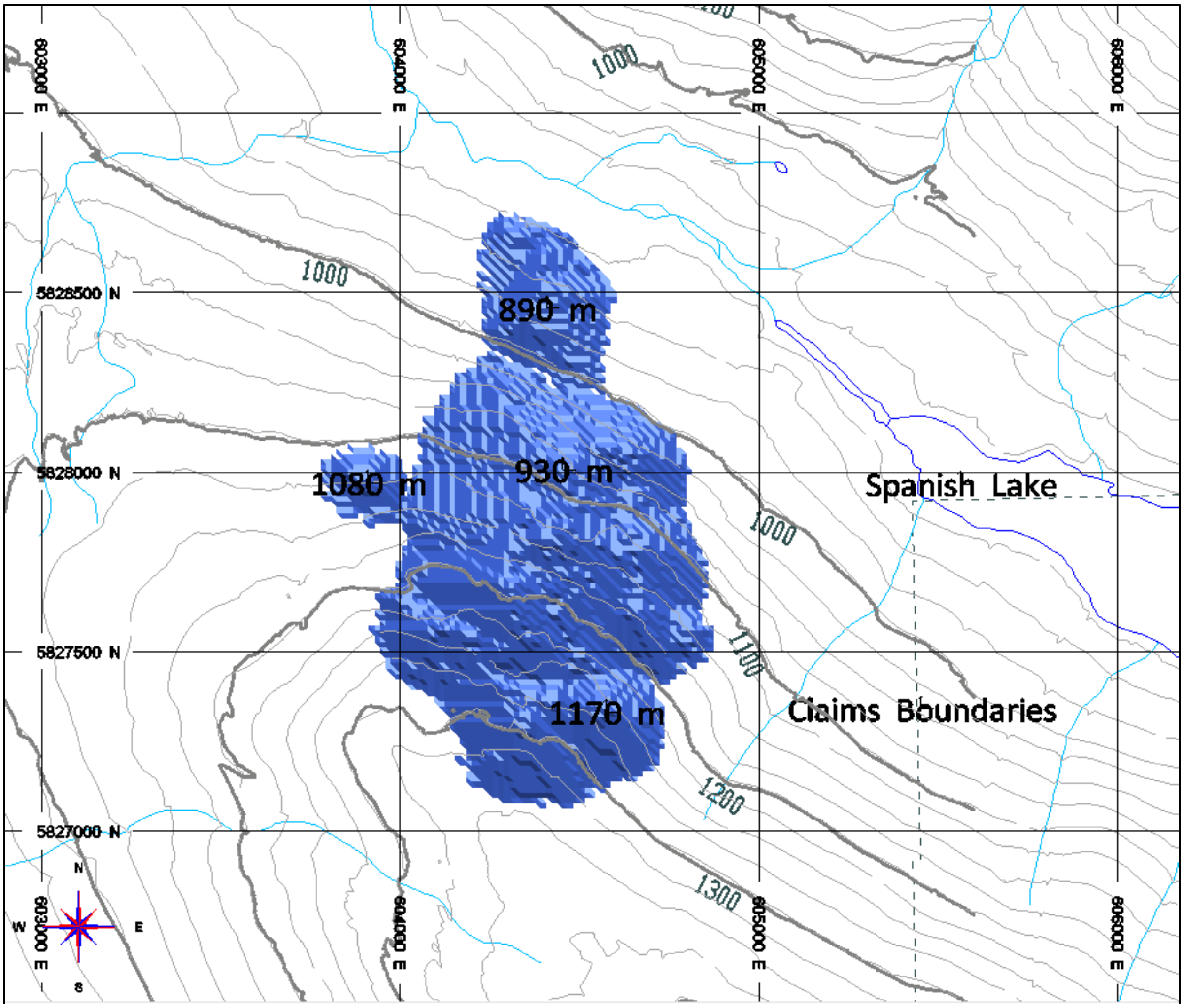


Figure 16-4 Plan View of Optimized Pit LG Shell

Block views show gold grade in all blocks above a 0.15 g/t cut-off. Inferred class blocks are shown with hatching. Green line represents original topography. All blocks within the Mineral Resource Bounding Shell are part of the Mineral Resource; blocks external to this shell are not part of the Mineral Resource.

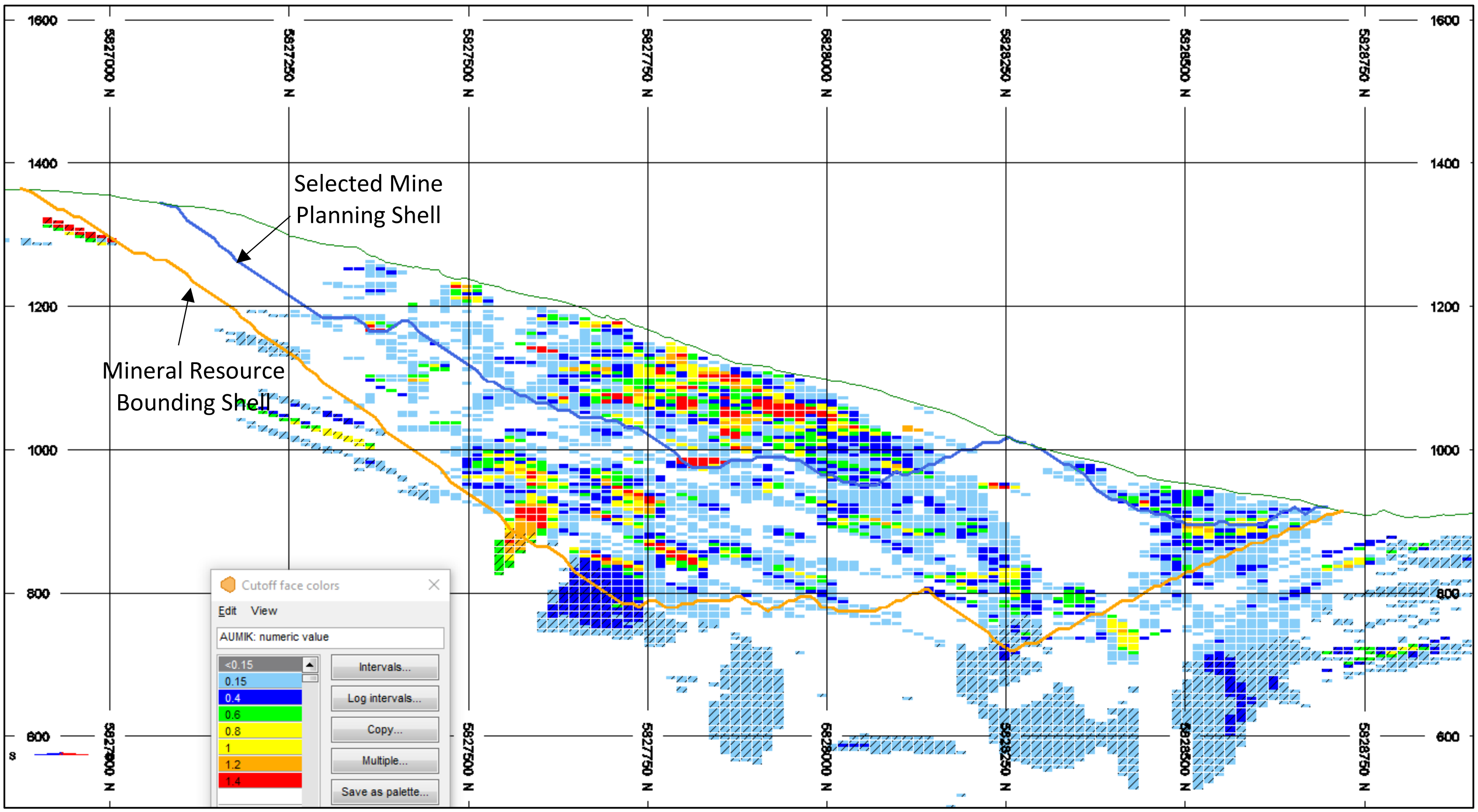


Figure 16-5 Cross Section View, 604385E (looking west), of Optimized Pit LG shell

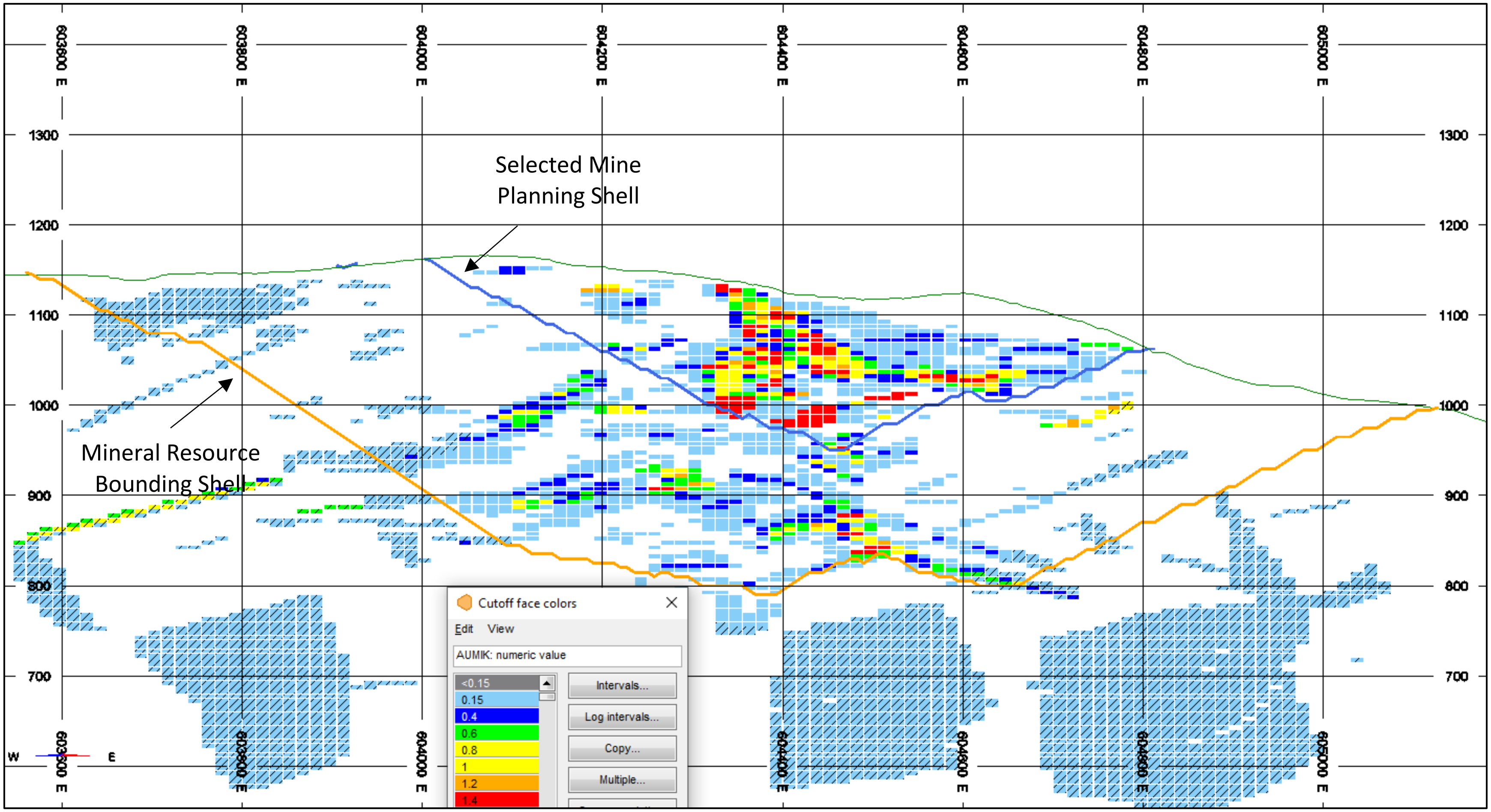


Figure 16-6 Cross Section View, 5,827,850N (looking north), of Optimized Pit LG shell

16.6 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 (7 million tonnes).
- The pit benches should be large enough to allow an efficient area for mining and keep the vertical bench advance rate to be <12 benches per year.
- Minimum mining width to allow an efficient area for mining is assumed to be 70 m.

The LG price cases described in the preceding sections can typically be used as a guideline for selecting interim pit phases. Pit shells created by the LG algorithm with lower input gold prices than the selected ultimate pit case will contain higher grade resources and/or lower strip ratios.
The LG shell generated using a 31% gold price factor is used to target a starter pit phase. The remaining phases are designed to push out to the optimized pit limits in various areas of the pit.

16.7 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 16-5 and Figure 16-2.
- Suitable single and dual lane haul road widths to handle 130 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 10 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.

16.7.1 Phase 1, P641

This designed pit contains just less than two years' worth of mill feed at a lower strip ratio and higher gold grade than the ultimate pit. This pit mines from the pit crest at the 1210 m elevation, down to the pit bottom at the 1040 m elevation via external roads.

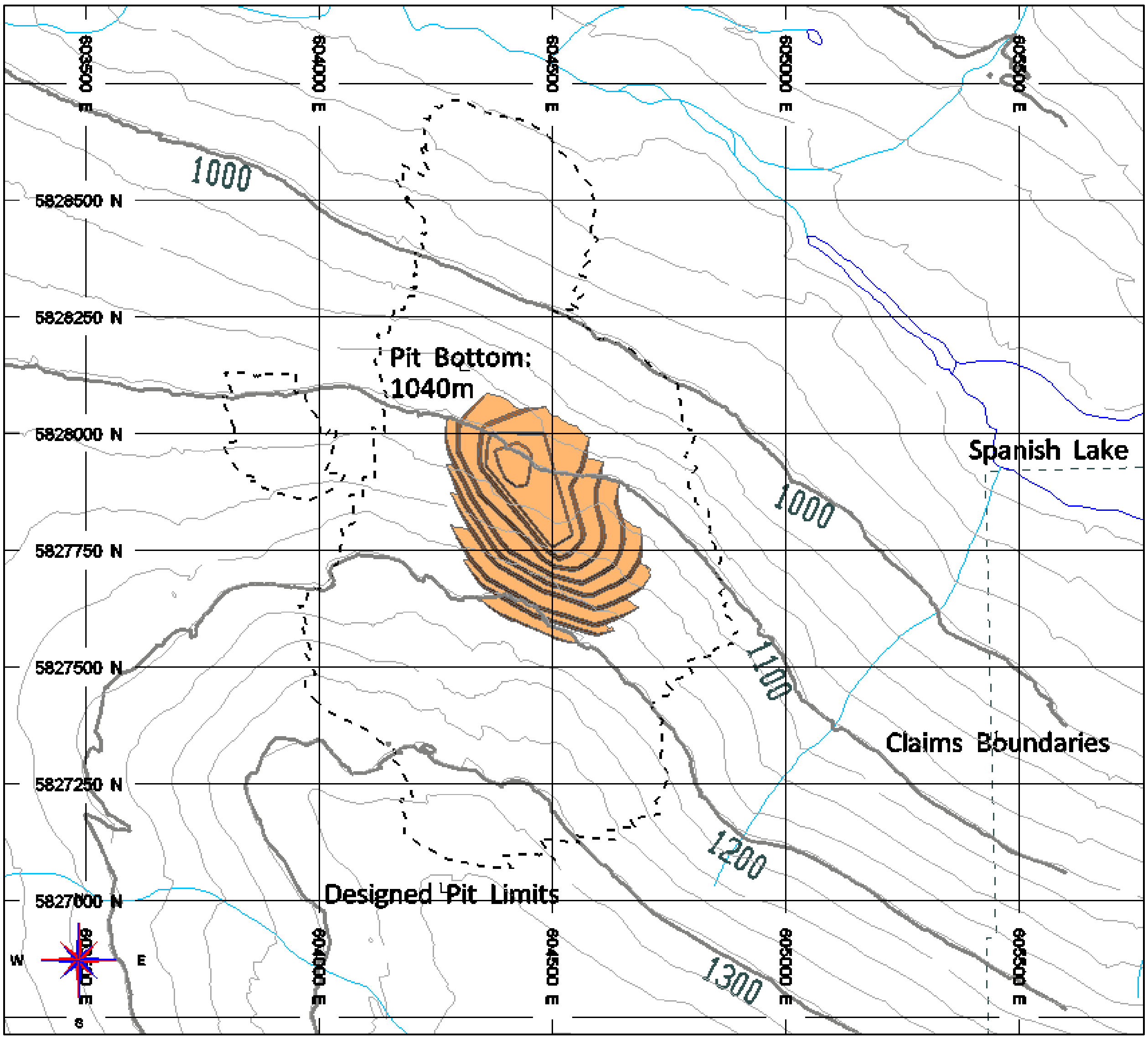


Figure 16-7 Plan View of Phase 1, P641

16.7.2 Phase 2, P642

Phase 2, P642 is a west pushback on the P641 pit. This pit mines from the pushback crest at 1255 m elevation down to the pit exit at the 1040 m elevation via external roads; then down the ramp to the pit bottom at the 1000 m elevation.

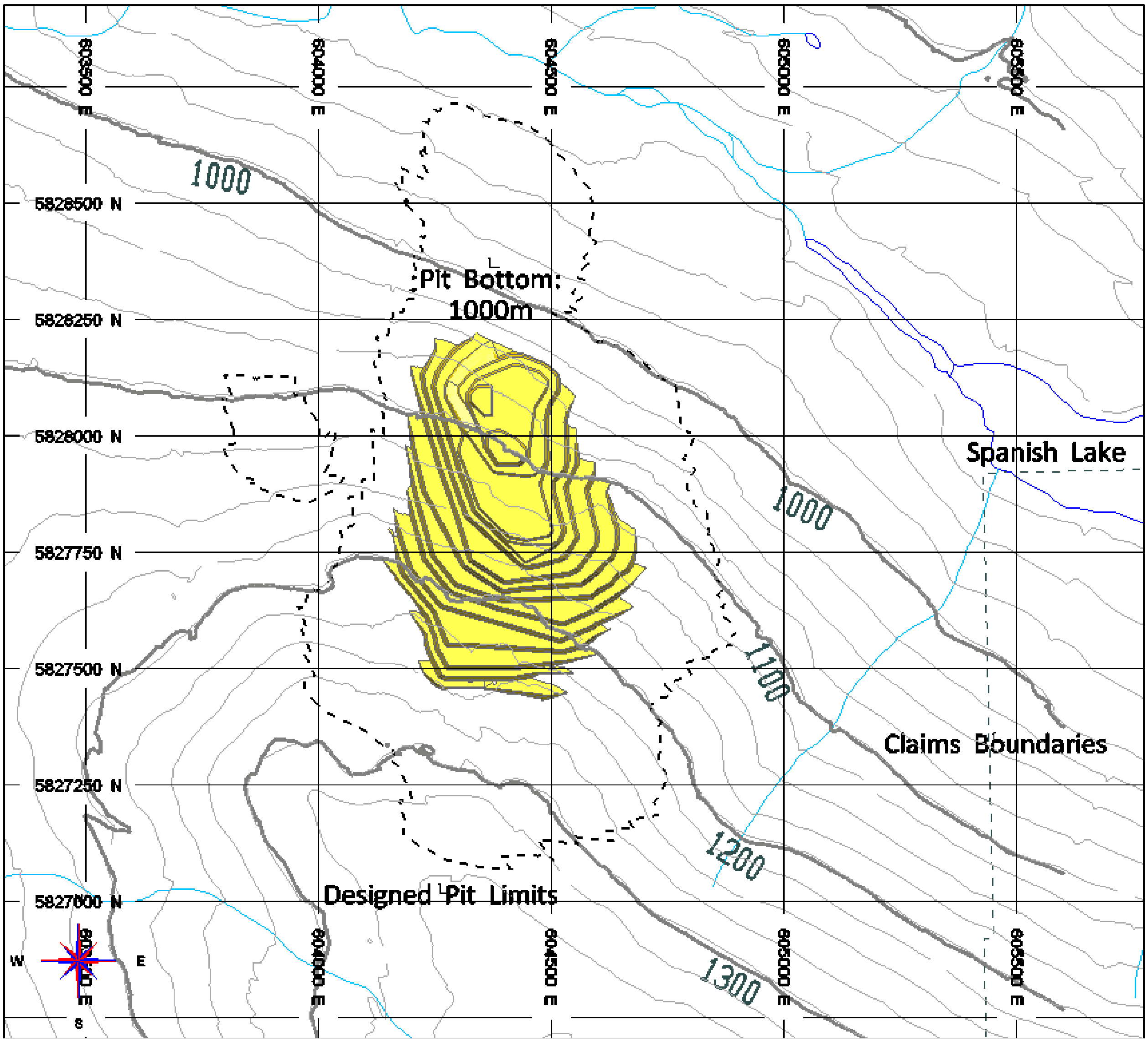


Figure 16-8 Plan View of Phase 2, P642

16.7.3 Phase 3, P643

Phase 3, P643 is a west, east and south pushback off the phase 2 pit to the designed pit limits. This pit mines from the crest at 1345 m elevation down to the pit exit at the 1030 m elevation via external roads; then down the ramp to the pit bottom at the 930 m elevation.

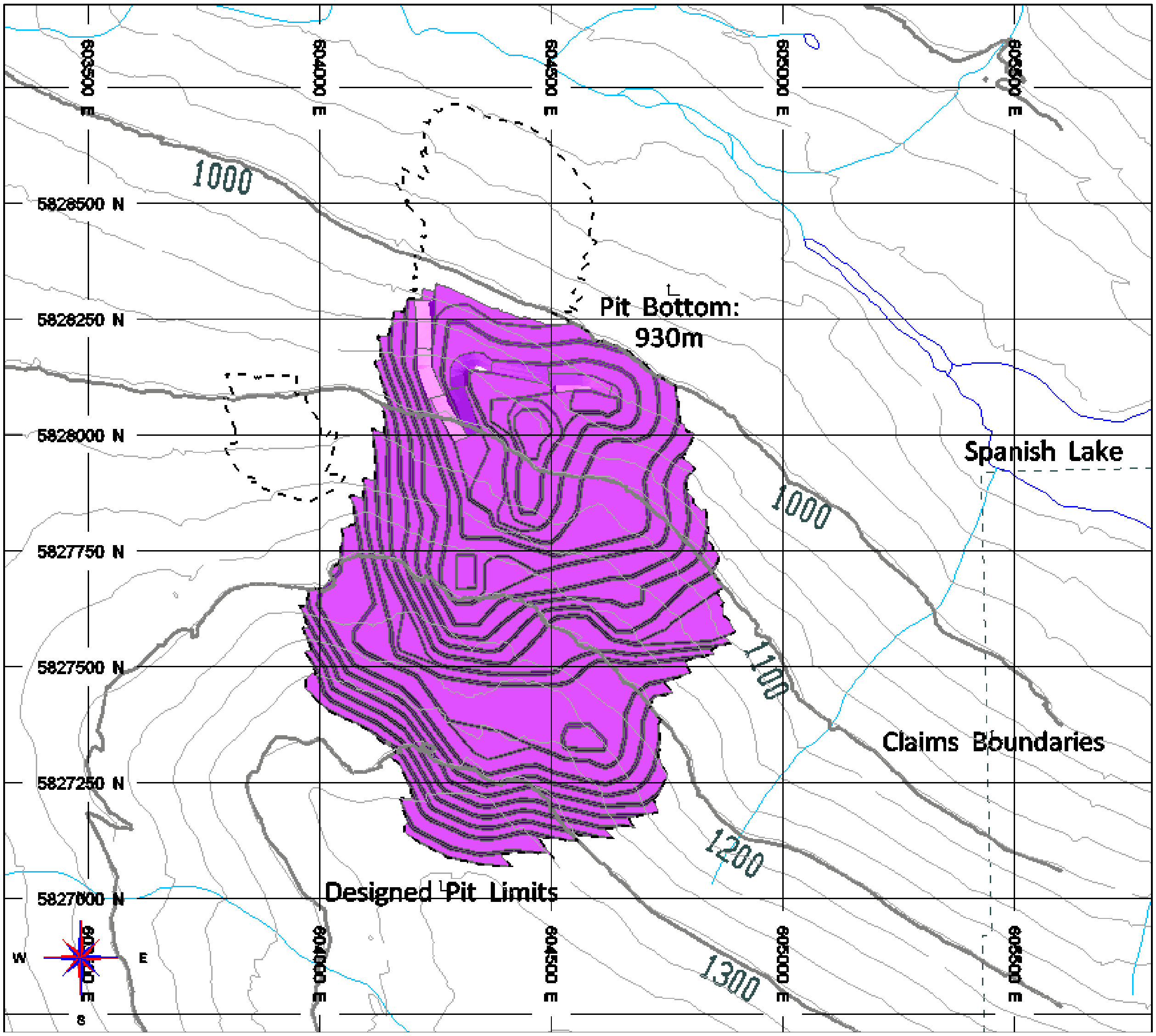


Figure 16-9 **Plan View of Phase 3, P643**

16.7.4 Phase 4, P644

Phase 4, P644 mines to the designed pit limits in the north portion of the deposit. A saddle is created between this phase and the phases to the south. This pit mines from the crest at 1030 m elevation down to the pit exit at the 960 m elevation via external roads; then down the ramp to the pit bottom at the 890 m elevation. This phase is mined independently of all other phases. Once mined out, this area can potentially be used for waste rock storage.

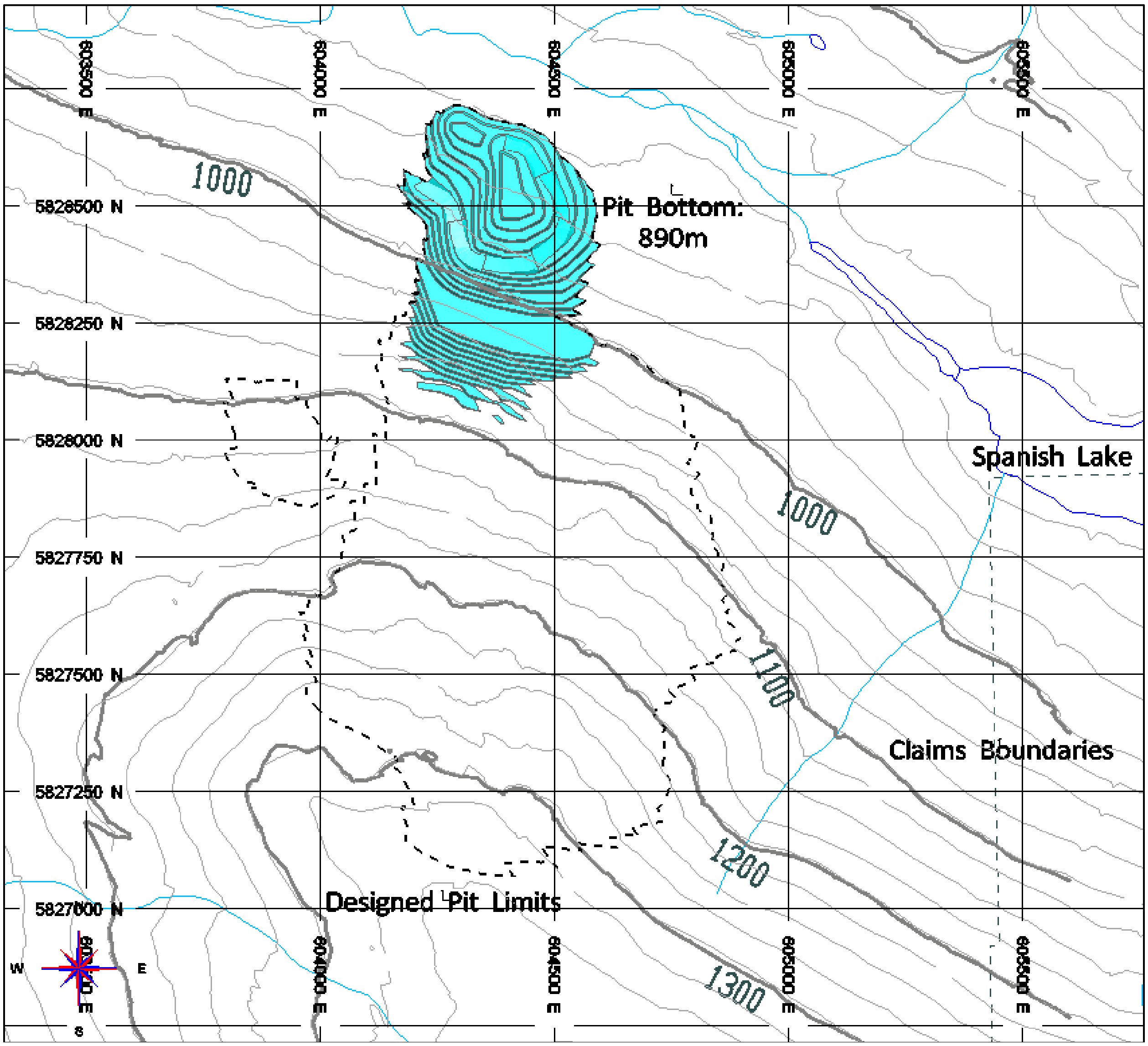


Figure 16-10 Plan View of Phase 4, P644

16.7.5 Phase 5, P645

Phase 5, P645 is a mini pit at the western edge of the designed pit limits. This pit mines from the crest at 1155 m elevation down to the pit bottom at the 1070 m elevation via external roads. This phase is mined independently of all other phases.

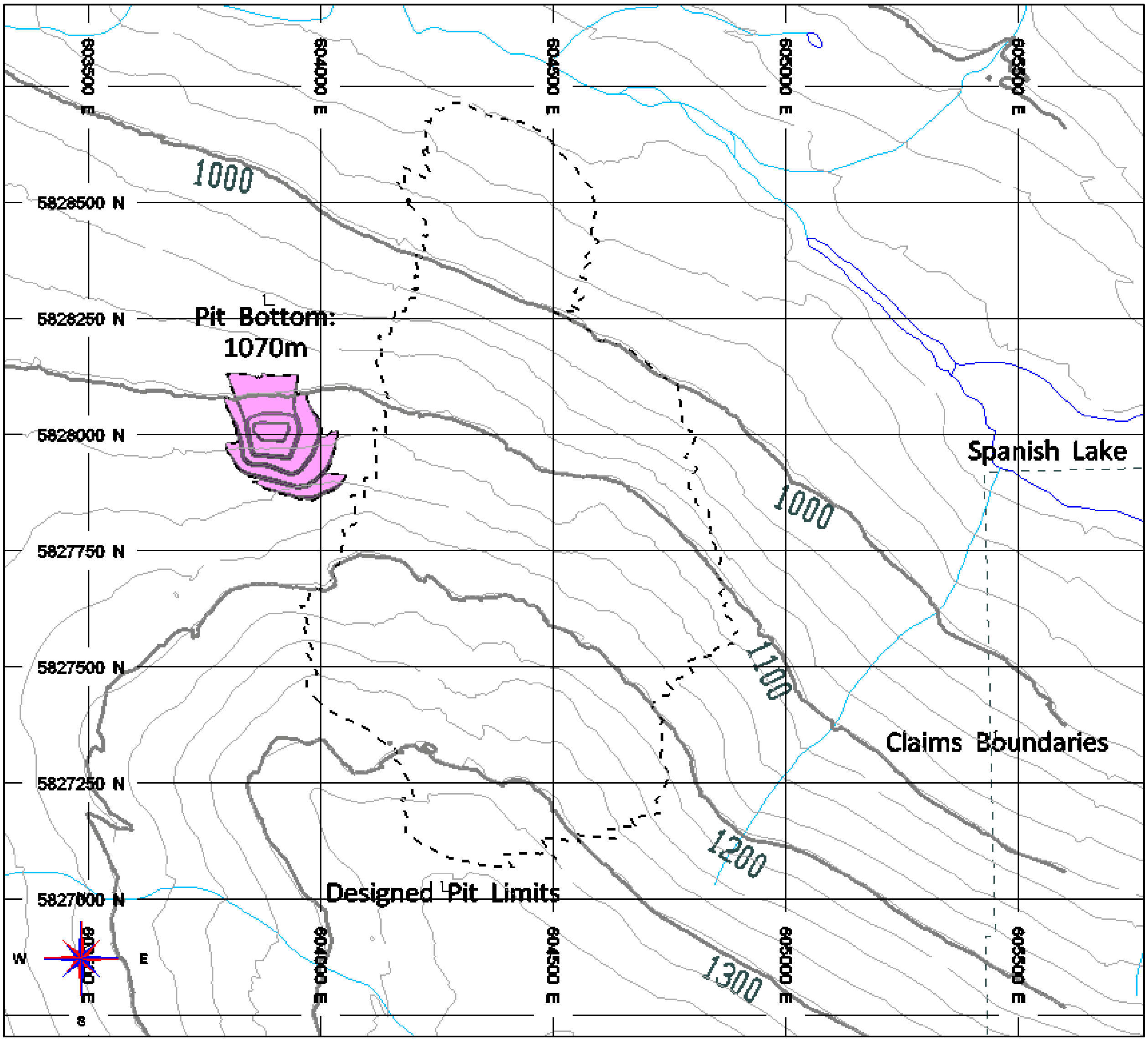


Figure 16-11 Plan View of Phase 5, P645

16.7.6 Ultimate Pit for Mine Planning

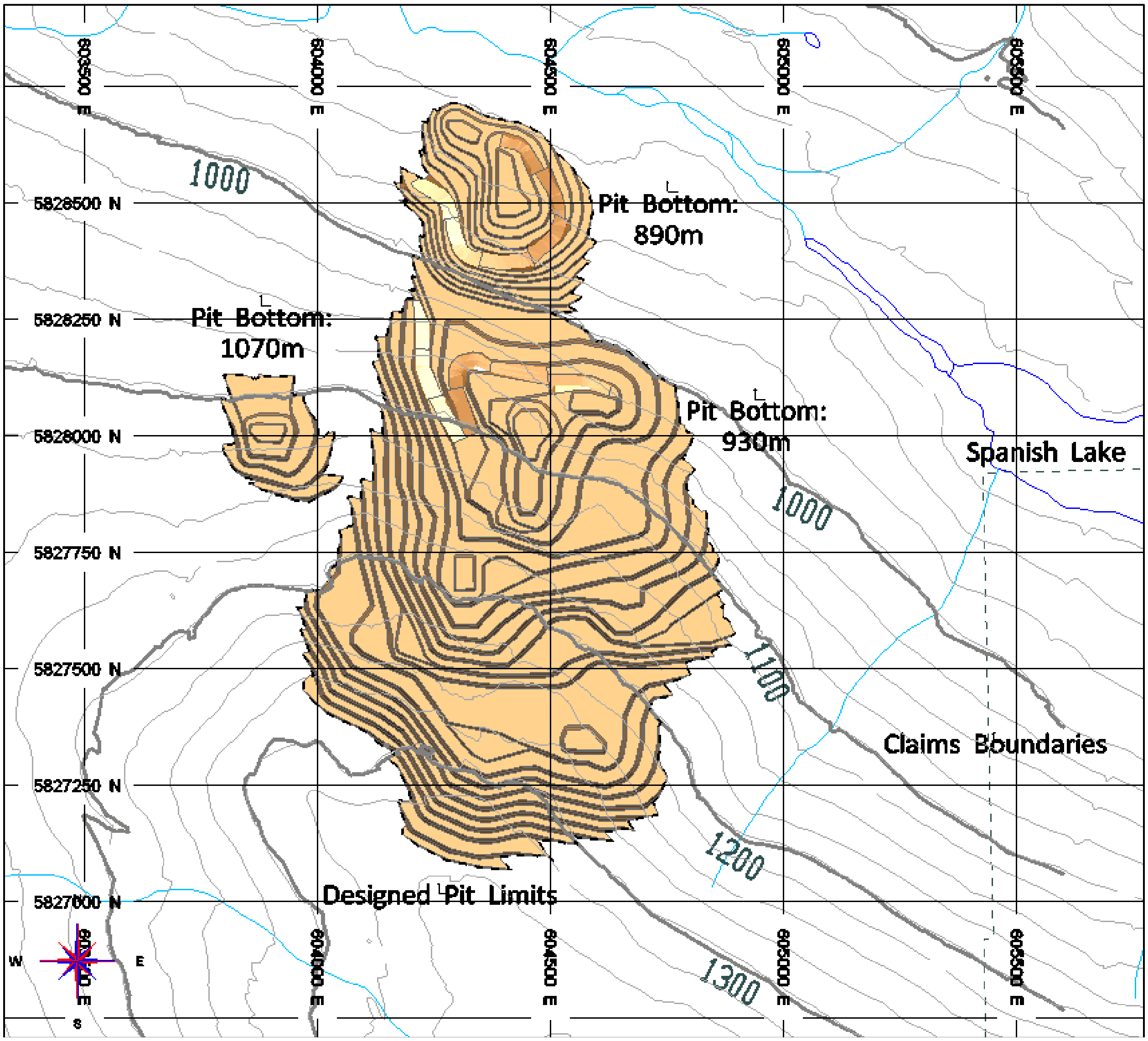


Figure 16-12 Plan View of Ultimate Pit for mine planning

Block views show gold grade in all blocks above a 0.15 g/t cut-off. Inferred class blocks are shown with hatching. Green line represents original topography. All blocks within the Mineral Resource Bounding Shell are part of the Mineral Resource; blocks external to this shell are not part of the Mineral Resource.

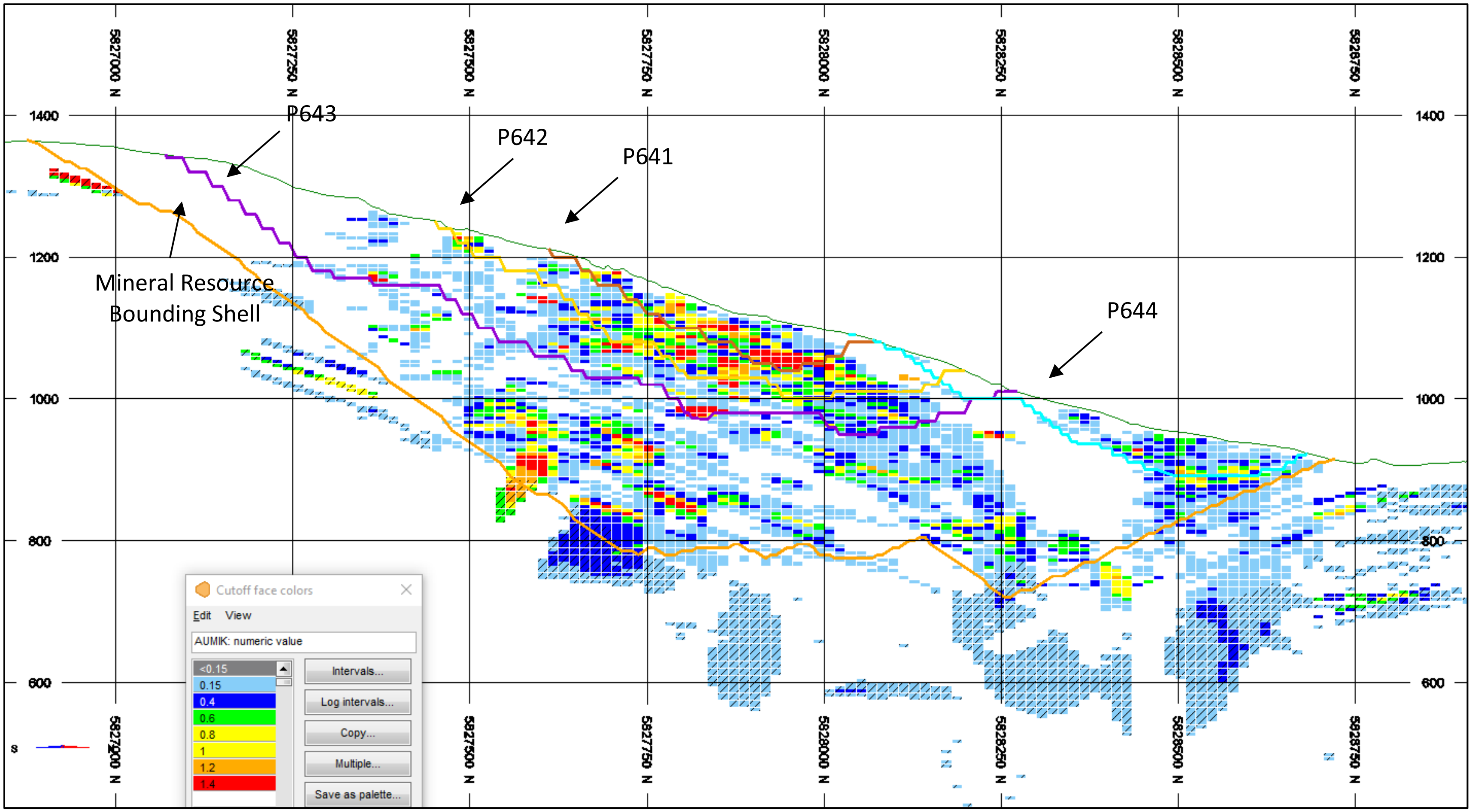


Figure 16-13 Cross Section View, 604385E (looking west) of Phased Pit Designs

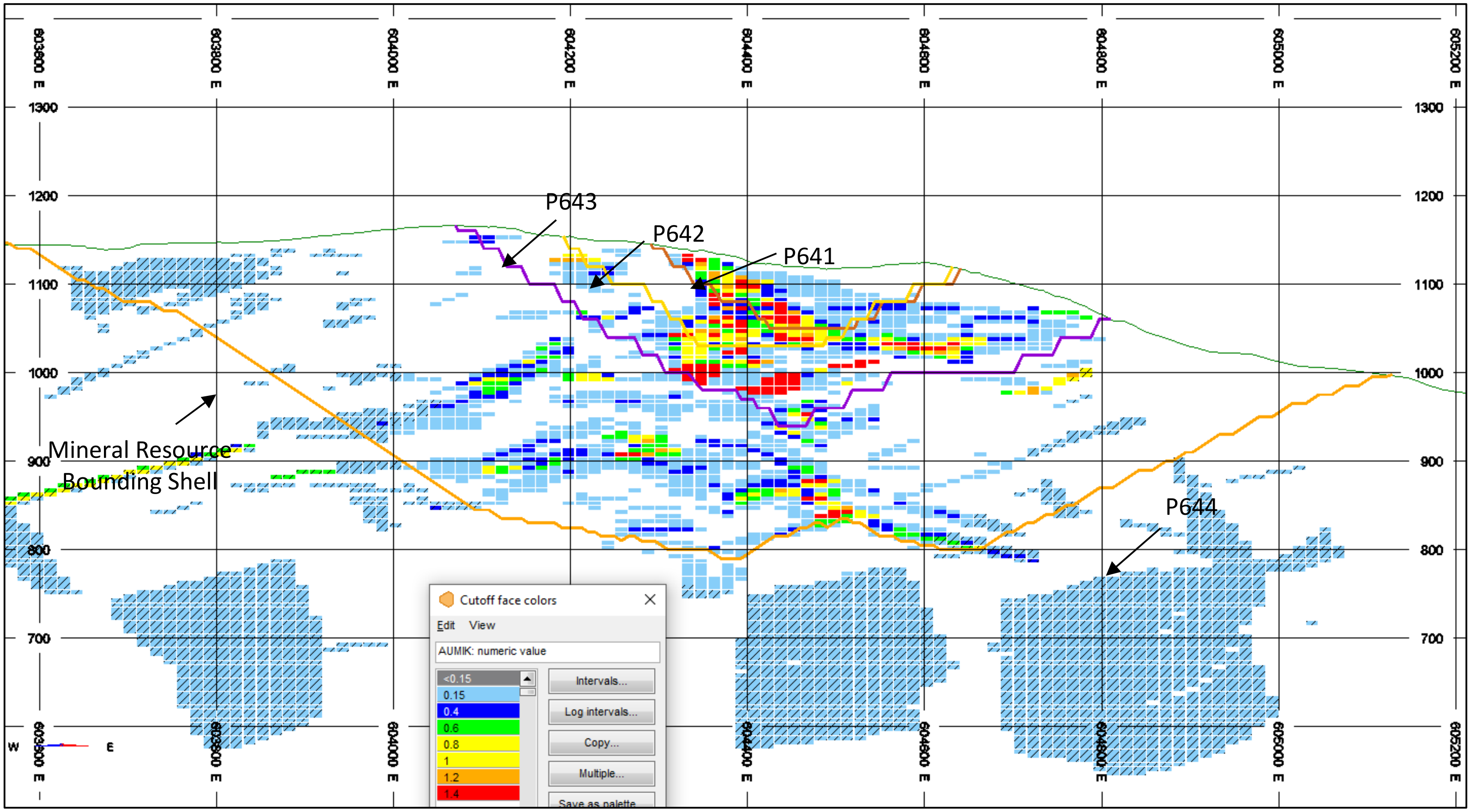


Figure 16-14 Cross Section View, 5,827,850N (looking north), of Phased Pit Designs

16.8 Pit Contents

The subset of the Mineral Resources delineated by the pit designs are shown in Table 16-7. The utilized cut-off gold grade is 0.40 g/t, and Inferred resources are treated as waste rock.

Table 16-7 Phased and Total Pit Delineated Resources

Pit Name	Units	P641	P642i	P643i	P644	P645i	Total
Measured Resource	kt	2,949	4,018	5,513	170	0	**12,650**
Gold Grade	g/t	0.95	1.04	0.91	0.61	0.00	**0.96**
Indicated Resource	kt	2,624	5,028	21,189	2,898	159	**31,898**
Gold Grade	g/t	1.02	0.89	0.95	0.67	1.52	**0.92**
Wasted Inferred Resource	kt	0	0	40	11	20	**71**
Gold Grade	g/t	0.00	0.00	1.62	0.44	0.94	**1.25**
Waste	kt	9,192	13,852	99,796	7,304	1,794	**15,646**
Mill Feed (M+I Resource)	kt	5,573	9,046	26,702	3,068	159	**44,548**
Gold Grade	g/t	0.98	0.96	0.94	0.67	1.52	**0.93**
Silver Grade	g/t	0.76	0.71	0.71	0.88	1.08	**0.73**
Waste	kt	9,192	13,852	99,836	7,315	1,814	**132,009**
Strip Ratio	Wst. / Res.	1.65	1.53	3.74	2.38	11.41	**2.96**
Total Pit Contents	kt	14,765	22,898	126,538	10,383	1,973	**176,557**

16.9 Waste Rock Management

16.9.1 Waste Rock Characterization

An acid rock drainage (ARD) potential criterion 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 16-15 shows the distribution of the waste rock ARD categories contained in the ultimate pit. Most waste is Ai (65%) and Bi (17%), 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.

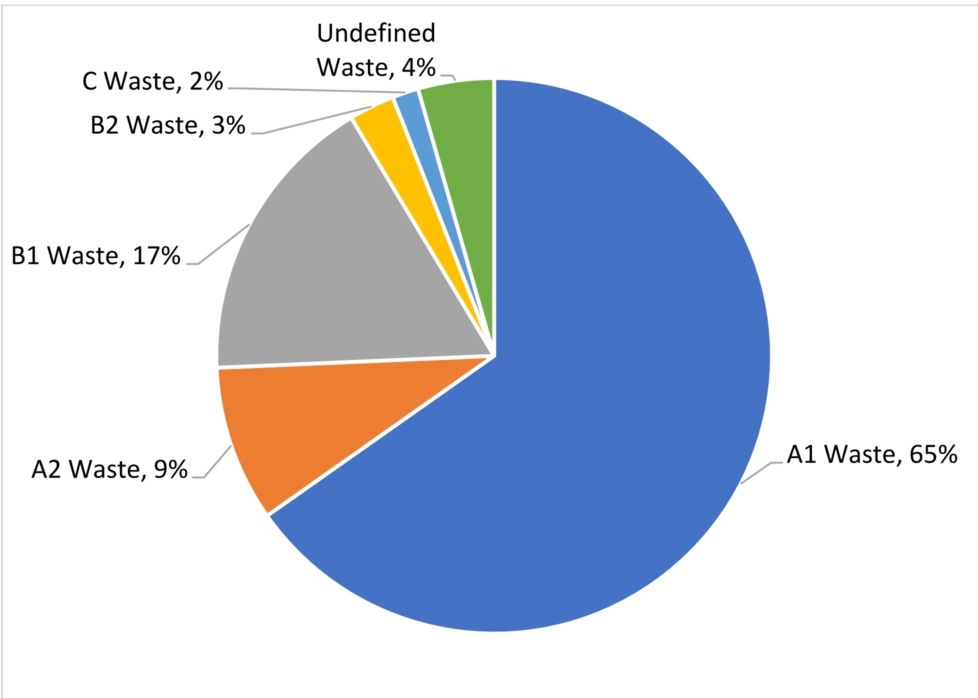


Figure 16-15 Pit Waste Rock Distribution by ARD Characterization

16.9.2 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 embankment
- 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, can be placed anywhere (e.g. WRSF's or tailings embankment).
- 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 similar to Aii.

16.9.3 **Waste Rock Disposal Strategy**

Suitable mine waste rock, Ai and Bi categories, 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 1.8 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 2.5:1 slope (22 degrees) from crest to toe.

Waste 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 1,190 m, containing 33 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 1,030 m, containing 13 Mt of waste rock.

Though there is also available space nearby for disposing waste rock on the east side of the pit, it is not considered for this study due to potential impacts on the drainages.

Suitable mine waste rock, Ai and Bi categories, will also be hauled to the tailing's facility and water management ponds for dam embankment construction, as required. It is estimated that 9.5 Mt of material will be required from the pit for the north and south dam embankments, as well as the water management pond through the LOM; including 2.5 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. It is therefore assumed that only 50% of Ai overburden and 37% of the undefined category of overburden will be suitable for the dam.

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, Bii, and C. A total of 25 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 the material will be submersed in less than two years.

Mineralized waste rock (between 0.15 and 0.40 g/t gold) 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 mineralized WRSF pile is 1,120 m, with a capacity of 48 Mt. The PEA 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 1-3.

16.10 Ex-Pit Haul Roads

Ex-pit haul roads are designed with a maximum grade of 10%. The roads will be built from waste rock fill from the pits. 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.

Preliminary ex-pit haul road layouts can be seen in Figure 1-3.

16.11 Resource 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 resources for later re-handle back the crusher.

A "High-Grade (HG)" stockpile is built to the south of the crusher, up to the 1,120 m elevation. This stockpile is planned to be partially reclaimed to the crusher once the open pit operations are completed.

Preliminary stockpile layouts can be seen in Figure 1-3.

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 resource material. 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. Mill feed and waste rock is defined in the blasted muck pile with a grade control system based on blasthole sampling, and a fleet management system keeps track of each load. 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 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 140 mm diameter holes will be used for production drilling. Three drills will be required. For this study, it is assumed that wall control will be established using buffer blasting techniques with the blasthole drill and a small diameter track drill will not be necessary. 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 preliminary 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 on-site, and the explosives plant will be housed in a secure structure. The plant and storage facilities will be located away from the mill site, pit, and all working areas, in compliance with regulatory requirements.

16.12.4 Loading

All resource material and waste rock require loading from the open pits into haul trucks. Loaders 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 two 12 m^3 sized hydraulic shovels, with the capability to excavate in backhoe configurations, will meet these requirements. One wheel loader with a 12 m^3 bucket is also specified for pre-production and waste stripping work. Loading units will also function to re-handle pit material, load overburden and topsoil, pit clean up, crusher support, road construction and snow removal. Crusher loading is planned to be done directly via hauler.

16.12.5 Hauling

All resource material 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 91-tonne maximum payload is targeted for this level of mine planning. The size of the fleet is determined by estimating the haulage productivities for mineralized and waste materials in each period. Haulage productivities are based on simulated hauler cycle times on representative haul routes. These cycle times includes loading, hauling, dumping, returning, all wait times and any inefficiencies in the hauling operation.

A secondary fleet of 38 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
- WRSF maintenance
- Ditching
- Reclamation
- Mobile Fleet fuel and lube support
- Open pit dewatering
- 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/sanding 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 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.

16.12.8 General Mine Expense and Technical Services

General mine expenses (GME) includes the supervision for the direct mining activities, including supervision for the mine fleet maintenance department. GME also includes the technical support requirements from Mine Engineering, Geology and Geotechnical functions.

16.12.9 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. It is assumed that several of these services will be combined in to shared buildings in future studies.

16.13 Mine Production Schedule

16.13.1 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 11 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 8.7 Mt of is required. The construction of the starter tailings dam will require 2.5 Mt during the pre-strip. Haul roads will require an additional 2.5 Mt during the pre-strip. The remainder is associated resource material that will be stockpiled, 1.0 Mt, overburden material that will be stockpiled, 2.4 Mt, and PAG rock that will be stockpiled, 0.3 Mt.

The top several benches in Phase 1 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 16-16 illustrates the mining activities to be completed by end of the pre-production period.

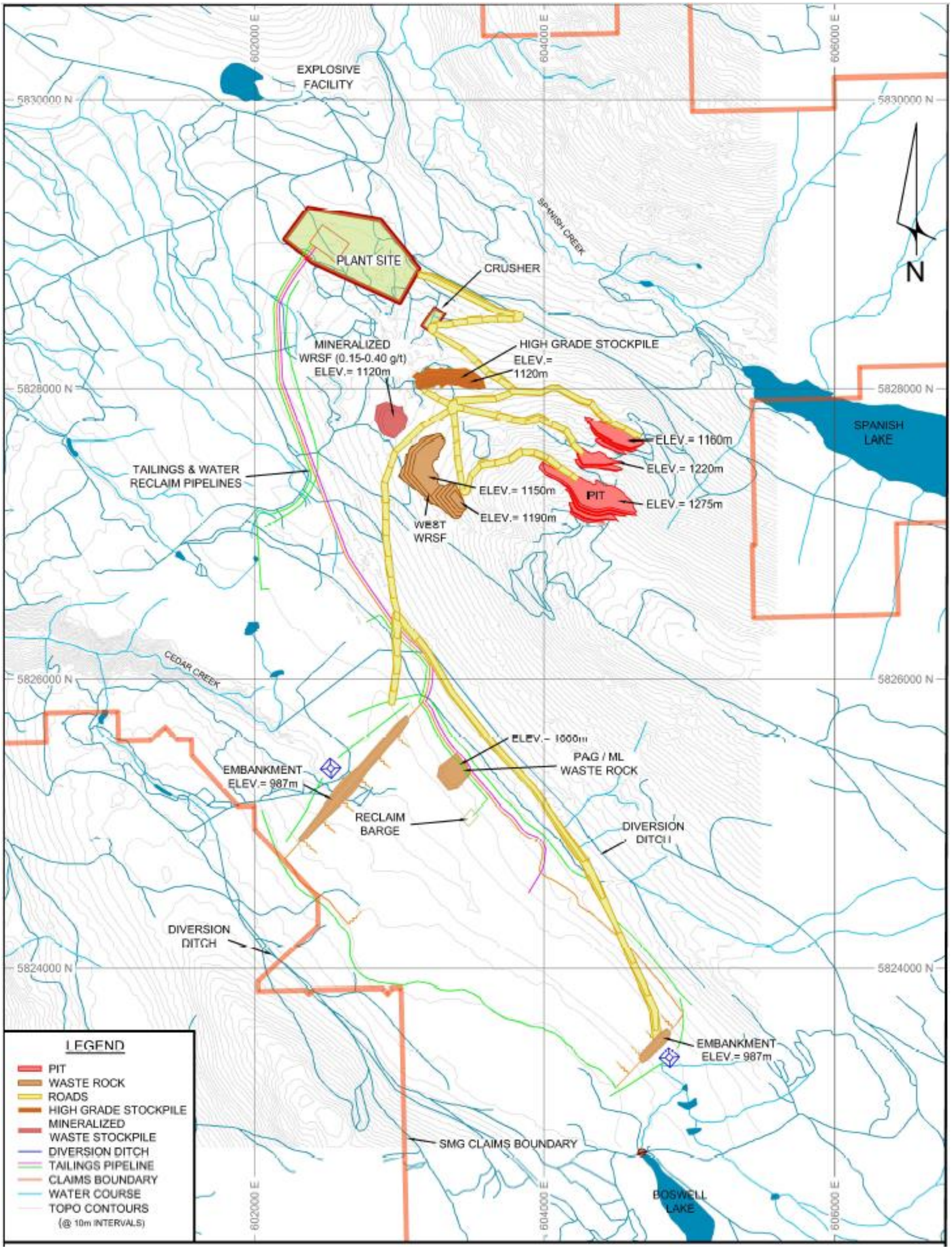


Figure 16-16 Mine Development - Pre-Production

16.13.2 Production Scheduling

The open pit mine production schedule is based on the following parameters:

- Annual mill feed of 3,650,000 t is targeted based on an average of 10,000 tonnes/day milling.
  - Mill production will ramp up through Year 1, totalling 3,000,000 t.
- Phased pit bench resources and waste rock contents are used as input to the mine production schedule (Table 16-7).
- 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.
  - Exceptions are P644 and P645 which are progressed independently of the other phases.
- Pit phase progression is limited to no more than eight benches each year. Average phase progression in an annual period is four benches.
- A cut-off grade strategy is employed. An elevated gold cut-off grade of 0.45 g/t is employed during the open pit operations. Only a portion of this stockpiled material is reclaimed.
- Material in the stockpiles is reclaimed to the mill after the pit is mined out in Y11.

The mine production schedule is shown in the following tables and graphs.

Table 16-8 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 Mined (kt)	Stockpile Wasted (kt)	Strip Ratio (t:t)	Total Mined (kt)	Total Moved (kt)
Y-2							1,921			1,921	**1,921**
Y-1		565					6,205			6,769	**6,769**
Y1	3,000	1,307		3,000	1.11	0.88	13,733		5.0	18,040	**18,040**
Y2	3,650	504		3,650	1.02	0.68	14,525		4.1	18,679	**18,679**
Y3	3,650	451		3,650	1.10	0.65	13,399		3.8	17,500	**17,500**
Y4	3,650	438		3,650	1.22	0.72	13,912		3.9	18,000	**18,000**
Y5	3,650	645		3,650	1.19	0.76	14,705		4.2	19,000	**19,000**
Y6	3,650	638		3,650	0.91	0.81	13,197		3.8	17,485	**17,485**
Y7	3,650	641		3,650	0.84	0.60	11,266		3.3	15,556	**15,556**
Y8	3,650	506		3,650	0.87	0.68	10,112		2.9	14,268	**14,268**
Y9	3,650	416		3,650	0.99	0.79	8,933		2.6	13,000	**13,000**
Y10	3,650	484		3,650	1.08	0.76	7,453		2.2	11,587	**11,587**
Y11	1,846	262	1,400	3,246	0.66	0.82	3,725	5,457	2.2	5,833	**7,233**
Total	**37,697**	**6,857**	**1,400**	**39,097**	**1.00**	**0.74**	**133,084**	**5,457**	**3.5**	**177,638**	**179,038**

Table 16-9 Pit to Mill of Stockpile Production Schedule, Phase Details

Period	Ex-Pit Const. Strip (kt)	P641, Starter Pit (kt)	P642, West Pushback (kt)	P643, South Pushback (kt)	P644, North Pit (kt)	P645, West Pit (kt)	Total (kt)
Y-2	690	11	17	1,202			**1,921**
Y-1		1,437	695	4,637			**6,769**
Y1		10,067	4,610	3,363			**18,040**
Y2		3,356	5,288	10,035			**18,679**
Y3			7,328	10,172			**17,500**
Y4			3,618	14,382			**18,000**
Y5			1,257	17,743			**19,000**
Y6			137	17,348			**17,485**
Y7				15,019	537		**15,556**
Y8				9,486	4,782		**14,268**
Y9				12,539		461	**13,000**
Y10				10,783	804		**11,587**
Y11					4,315	1,518	**5,833**
Total	**690**	**14,871**	**22,950**	**126,708**	**10,438**	**1,980**	**177,638**

Table 16-10 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)	Low Grade (0.15 to 0.40 g/t gold) Stockpile (kt)	Inferred Stockpile (kt)	High Grade (> 0.40 g/t gold) Stockpile (kt)
Y-2	1,919						1		
Y-1	1,599	2,400	278			1,476	418	32	565
Y1	1,172	1,400	2,783	2,250		1,098	4,831	188	1,307
Y2		550	2,198			6,621	4,861	287	504
Y3		550	3,697		200	5,098	3,820	31	451
Y4		550	3,248			6,169	3,926	7	438
Y5		550	2,278			6,336	5,523		645
Y6		550	3,567			3,450	5,619		638
Y7		500	1,655		2,700	735	5,661		641
Y8			2,125		4,190		3,702	81	506
Y9			1,644		3,600	381	3,291	11	416
Y10			1,224		2,320		3,902		484
Y11			371		350	1,118	1,806	70	262
Total	**4,690**	**7,050**	**25,071**	**2,250**	**13,360**	**32,481**	**47,360**	**706**	**6,857**

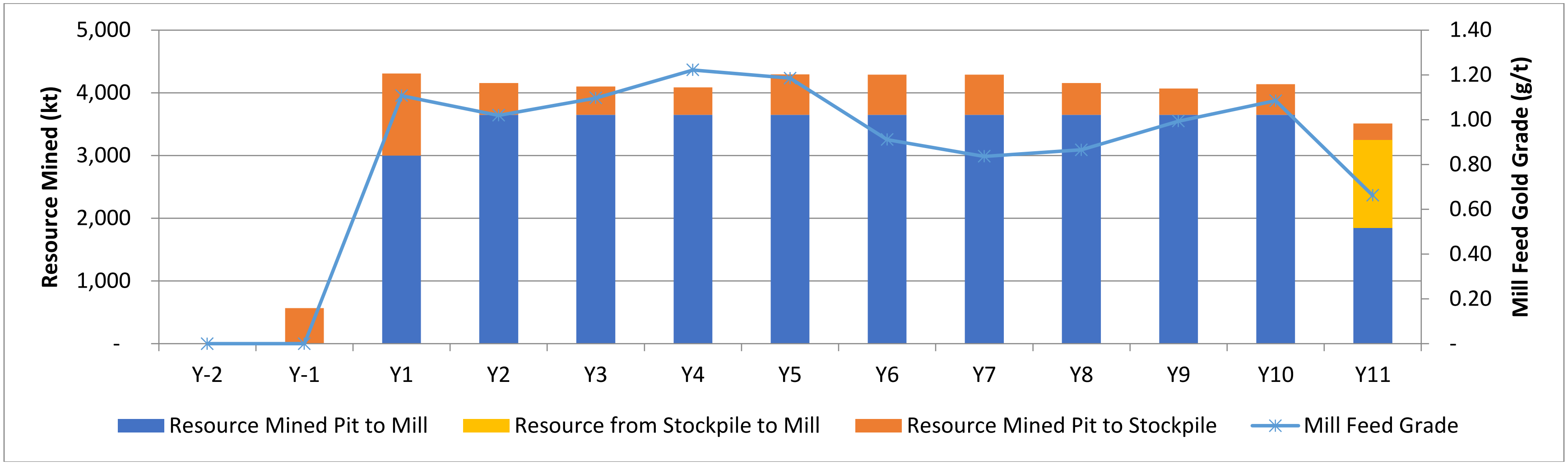


Figure 16-17 Mine Production Schedule, Resource Mined and Mill Feed Grades

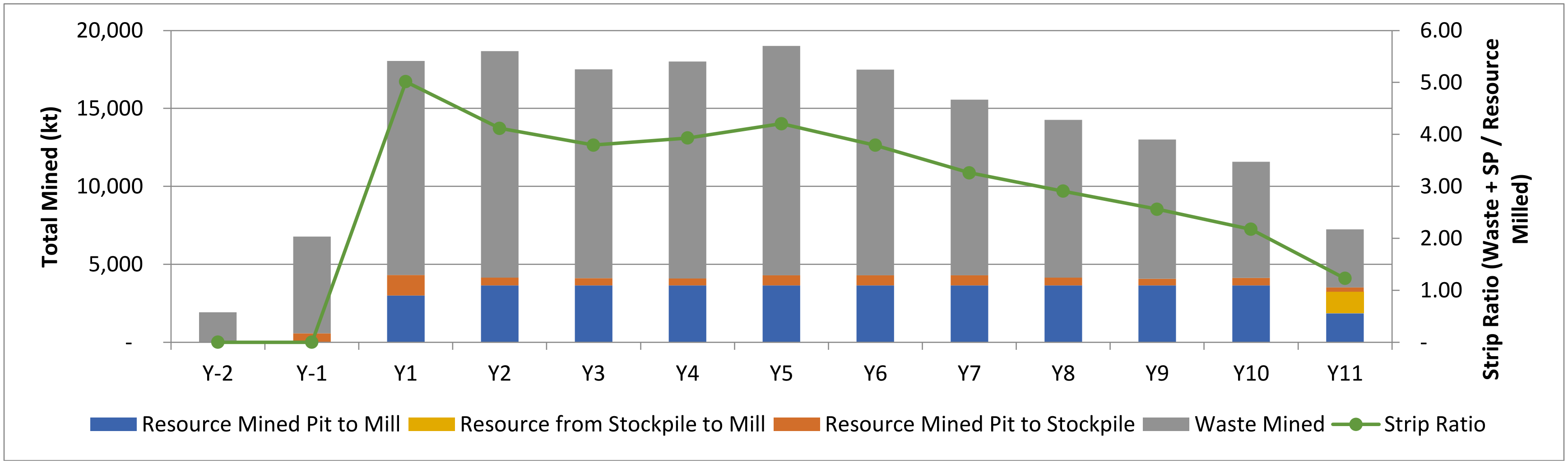


Figure 16-18 Mine Production Schedule, Total Mined and Strip Ratio

16.14 Mine End of Period Maps

The following figures show the general arrangement of the mine operations at Year 1, Year 2, Year 6 and Year 11, the completion of open pit operations.

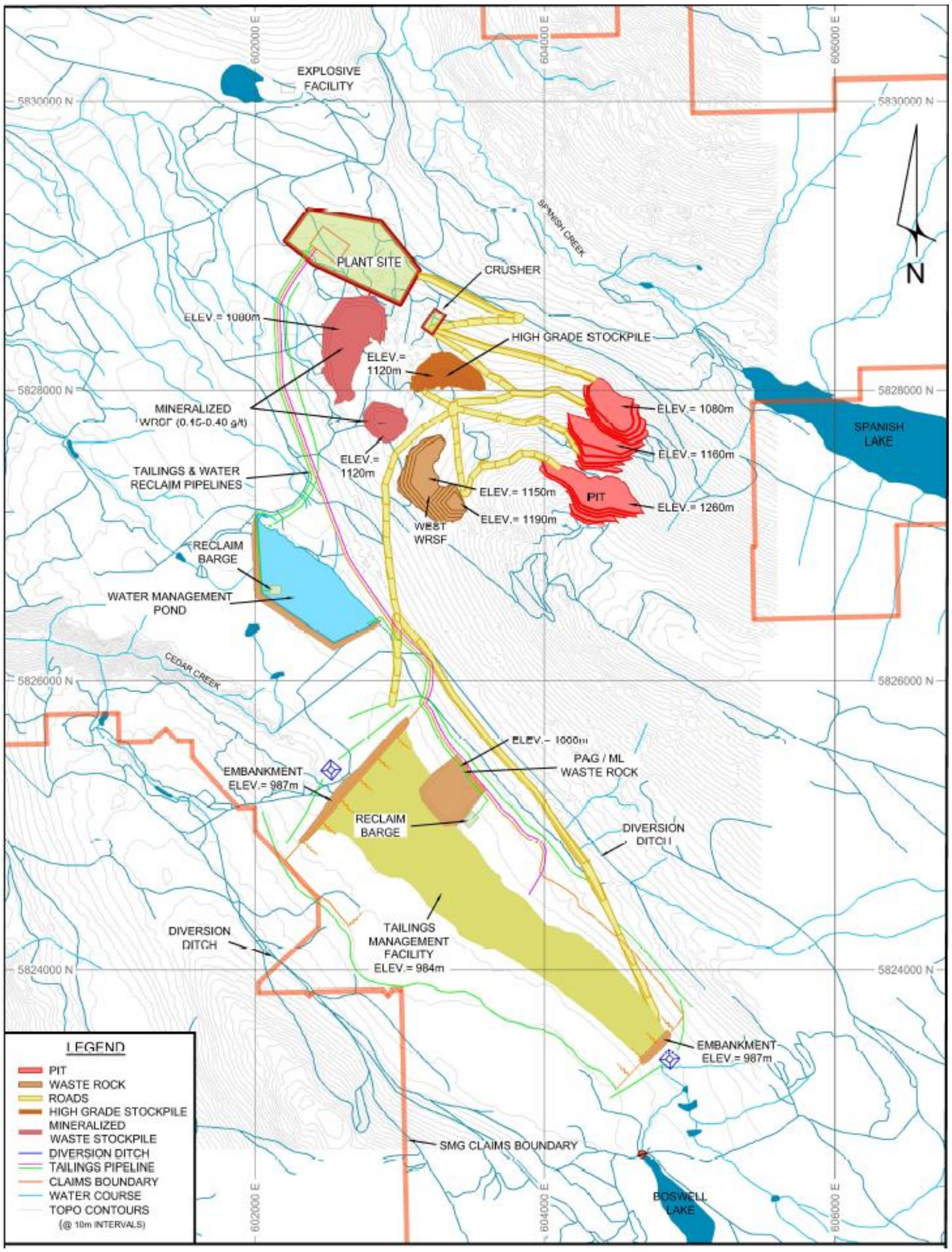


Figure 16-19 Year 1 End of Period Map

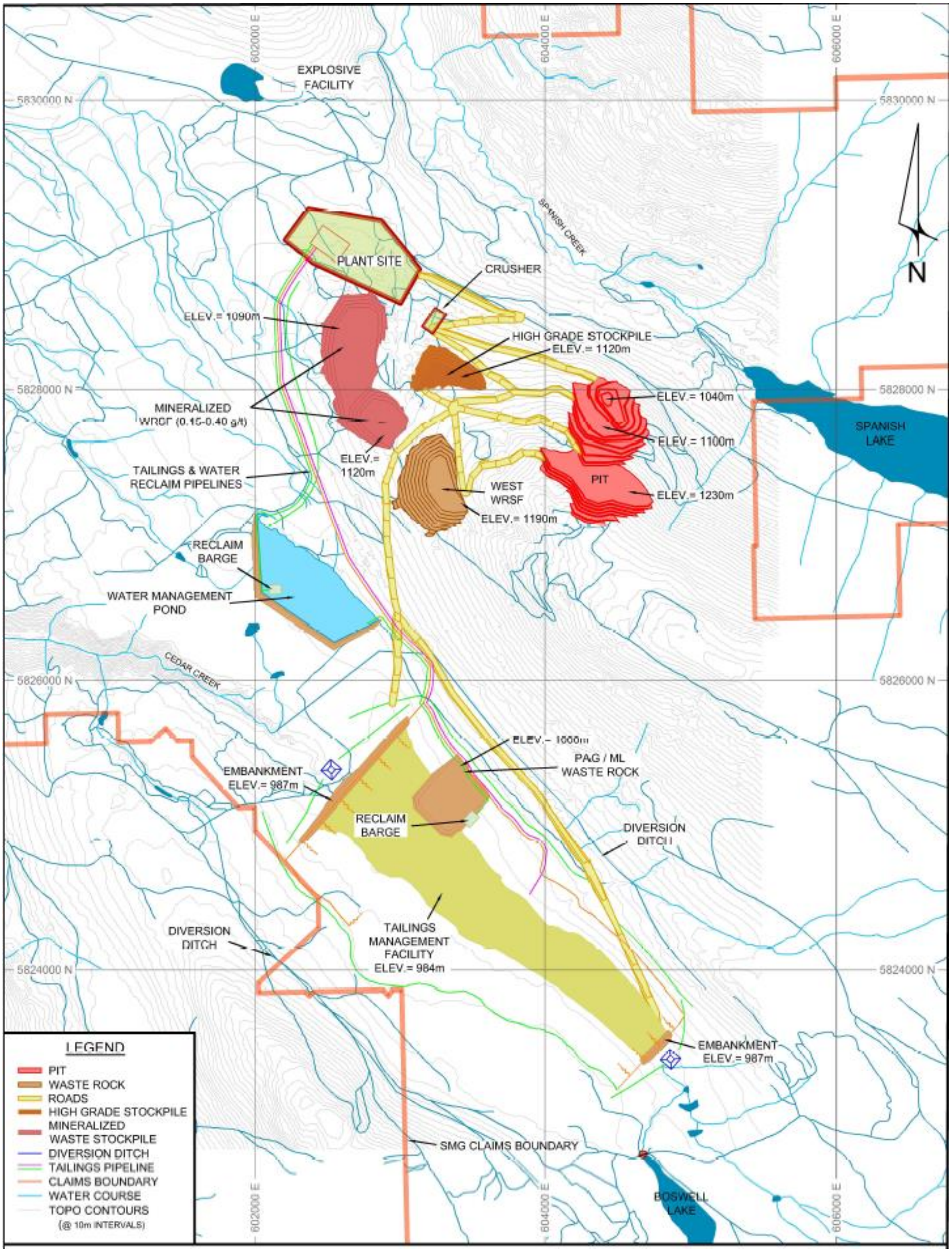


Figure 16-20 Year 2 End of Period Map

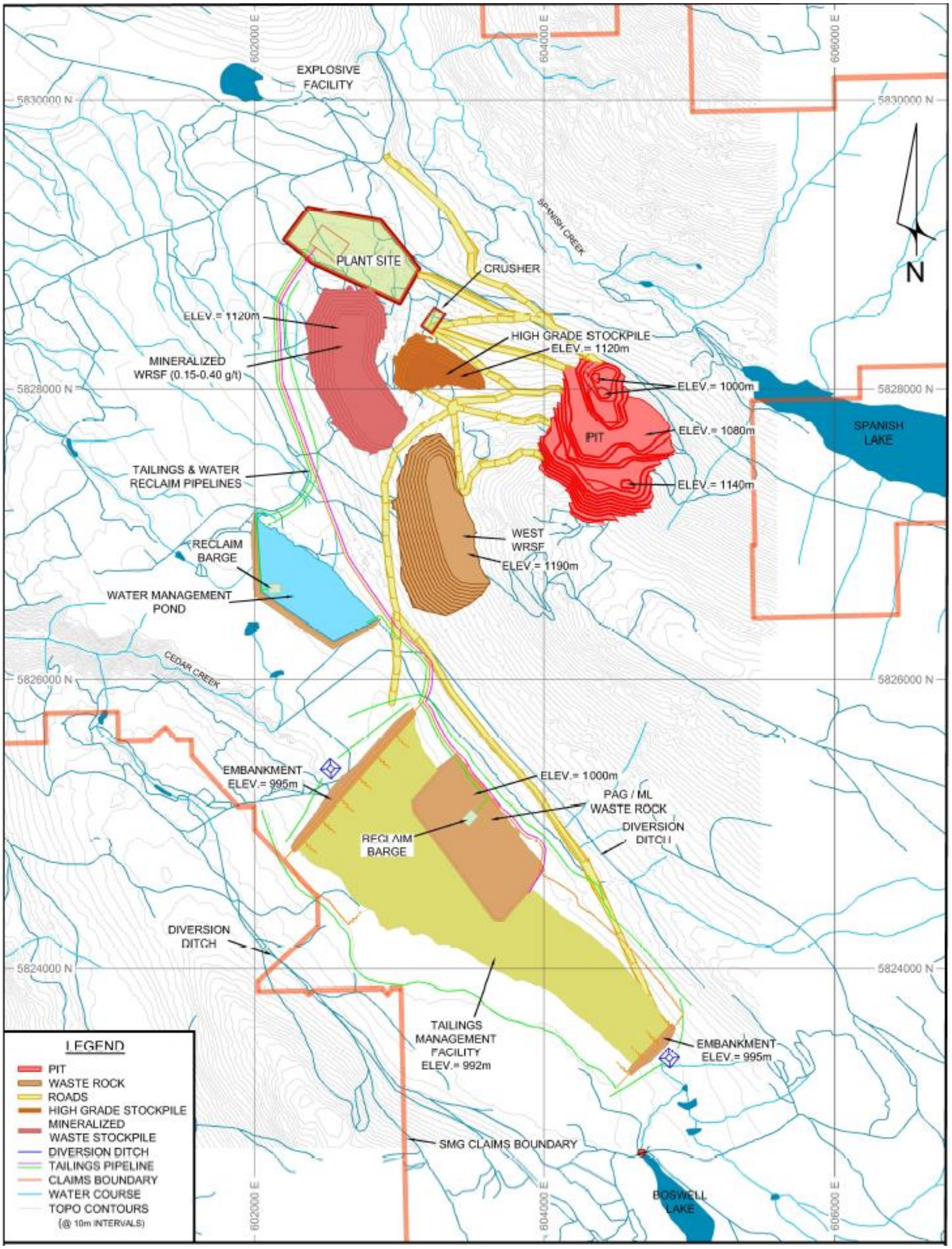


Figure 16-21 Year 6 End of Period Map

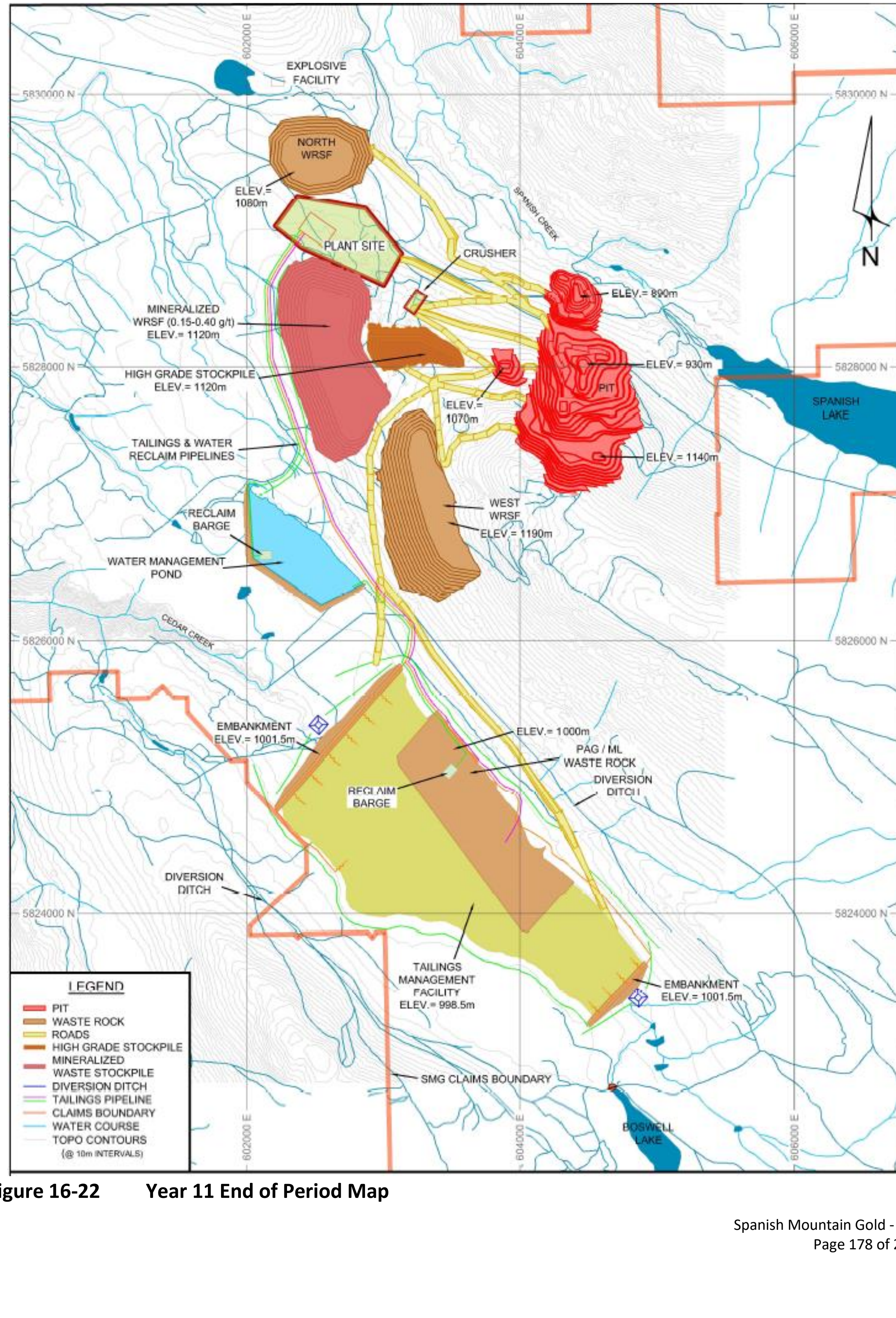


Figure 16-22 Year 11 End of Period Map

17.0 Recovery Methods

17.1 Process Flowsheet

Metallurgical test work results discussed in Section 13 confirm that ROM ore from the Spanish Mountain Gold (SMG) deposit can be processed using:

- Crushing;
- Grinding;
- Flotation;
- Gravity concentration of cleaner and re-cleaner flotation tails;
- Carbon-in-Leach (CIL) of combined flotation and gravity concentrates;
- Carbon elution and Electro-winning;
- and Cyanide destruction of CIL residues with the SO_2/Air process.

Unit processes selected for the design of the process plant are based on the results of metallurgical testing described in Section 13. The metallurgical process selected for the PEA produces gold-silver doré as a final product.

A simplified process flowsheet is shown in Figure 17-1.

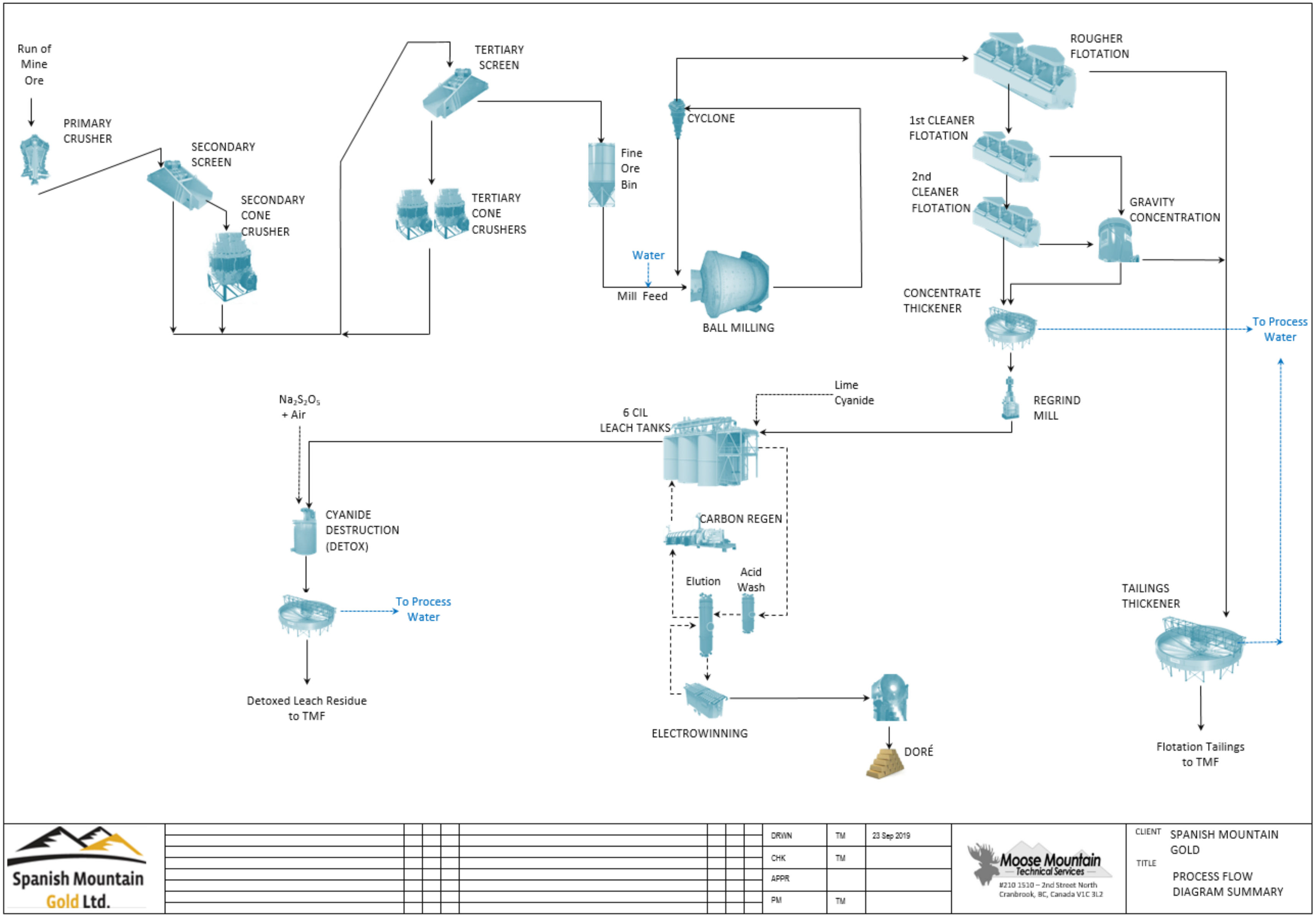


Figure 17-1 Simplified Process Flowsheet for 10,000 t/d

17.2 Major Design Criteria

The concentrator has been designed to treat gold-bearing material at the rate of 10,000 t/d. The major design criteria are outlined in Table 17-1.

Table 17-1 Major Design Criteria

Description	Unit	Value
Mill Feed Throughput	tpa	3,650,000
Operations		
Crusher Availability	%	73
Plant Availability	%	92
Plant Daily Throughput	tpd	10,000
Plant Hourly Capacity	tph	453
Average ROM Feed Au Grade	g/t	1.00
Max ROM Feed Au Grade	g/t	1.50
Crushing		
Crusher Work Index	kWh/t	9.5
Primary	type	Jaw
Secondary	type	Cone
Tertiary	type	Cone
Fine Ore Bin Capacity	Tonnes	10,000
Grinding		
Bond work index	kWh/t	12.2
Ball Mill 1 Dimensions	Dia ft x EGL ft	16.5 x 30
Ball Mill 1 Power	kW	3,700
Mill Feed Particle Size F_{80}	mm	12.0
Mill Product Particle Size P_{80}	µm	184
Mill Classification	type	Cyclones
Rougher Flotation		
Residence Time	min	18
Number of Cells	number	9
Cell Volume	m^3	40
Cleaner Flotation		
Residence Time	min	13
Number of Cells	number	3
Cell Volume	m^3	18
Re Cleaner Flotation		
Residence Time	min	13
Number of Cells	number	3
Cell Volume	m^3	18
Gravity Concentration		
Gravity Concentration	type	Sepro SB5200

Description	Unit	Value
Regrind		
Concentrate Production	% of mill feed	3.4
Concentrate Regrind Mill	type	Vertical
Concentrate Regrind Mill Power	kW	200
Regrind Product P80	µm	35
CIL and Carbon Desorption		
Residence Time	Hours	48
Number of Tanks	number	6
Tank Diameter	m	6.2
Tank Height	m	7.8
Carbon Concentration	g/L	15-20
Cyanide Destruction		
Method	type	SO_2 Air
Reagent	type	$Na_2S_2O_5$
Reagent addition	kg/t mill feed	0.6
Final Cyanide Target (WAD)	mg/L	< 0.2
Tailings Thickener		
Thickener U/F density	%	65%
Thickener Diameter	m	30

17.3 Process Operating Description

The 10,000 t/d process plant flowsheet design uses a conventional process technology.

17.3.1 Crushing

Crushing will be carried out using a three-stage crushing circuit with a capacity of 595 tph and availability of 73%. The crushing circuit will include:

- a primary jaw crusher;
- a secondary cone crusher and single deck screen;
- and a tertiary cone crusher and single deck screen;

Run of mine ore will be hauled to the primary crusher using 90 tonne payload haul trucks. The trucks will dump onto a static grizzly. The primary jaw crusher will operate in open circuit with a closed size setting (CSS) of 175 mm. A tramp magnet removes steel from the primary crushed ore conveyor before the secondary crushing stage.

The secondary cone crushing station operates in open circuit with a CSS of 55 mm and a pre-classification screen.

Ore from the secondary crusher is conveyed to the tertiary crushing stage which operates in close-circuit using two short head cone crusher stations with pre-classification vibrating screens.

Final crushed product size with P_{80} of 12 mm is conveyed to a 20,000 tonne fine ore bin.

17.3.1 Grinding

Grinding from a P_{80} of 12 mm to P_{80} of 180 µm is carried out by ball mill in a closed circuit with a cyclone. The grinding circuit can process a nominal 10,000 tpd at 453 tph and 92% availability and 250% recirculating load. The ball mill is 16.5 feet diameter x 30 feet length mill with 3,700 kW motor.

17.3.2 Flotation

Cyclone overflow from the grinding circuit is pumped to a flotation conditioning tank. Potassium amyl xanthate (PAX) is used as a general-purpose flotation collector. Methyl isobutyl carbinol (MIBC) is used as a frother.

Rougher flotation is carried out in nine conventional 40 m^3 mechanical cells, each using forced-air. Rougher flotation tails are pumped to the tailings thickener.

Rougher concentrate gravity flows to cleaner and recleaner flotation where Carboxymethylcellulose (CMC) is used to depress organic carbon. Recleaner concentrate is pumped to a concentrate thickener.

Cleaner and recleaner tails gravity flow to a semi-batch gravity centrifugal concentrator. Gravity tails are pumped to the tailings thickener. Gravity concentrate is transported to the concentrate thickener.

Thickener overflow gravity flows to a tank where it is recycled for plant use. Thickener underflow is pumped at 40% solids to a 200 kW vertical regrind mill. Regrind product at approximately P_{80} 35 µm is pumped to the CIL for leaching.

17.3.3 CIL

Leach feed from the regrind mill is first pumped to a CIL feed sampler, and then slurry is contacted with carbon using six CIL tanks operating in series accounting to a total of 48 hours of residence time. Sodium cyanide and lime slurry is added to CIL Tanks 1 and 3.

Carbon concentrations of 20 g/L are required in all tanks. Barren carbon enters the adsorption circuit at CIL Tank 6 and moves countercurrent to the slurry flow using interstage screens and pumps from downstream to upstream tanks.

The countercurrent process is repeated until the carbon becomes loaded and reaches CIL Tank 1. Carbon is then moved to a loaded carbon recovery screen. The loaded carbon is washed with water and pumped to the desorption area. Underflow from the loaded carbon recovery screen is returned to CIL Tank 1.

The slurry from CIL Tank 6 flows by gravity to a carbon safety screen to recover any carbon in the event of damage to the CIL Tank 6 interstage screen. Recovered carbon is collected in a bin for manual transfer.

Underflow from the carbon safety screen gravitates to a cyanide destruction tank.

17.3.4 Carbon Desorption and Regeneration

Carbon desorption and regeneration is carried out by acid washing of carbon, stripping of gold from loaded carbon (elution), and carbon regeneration.

Carbon from the loaded carbon screen is pumped to an acid wash. Acid wash is carried out with dilute hydrochloric acid with an acid wash column inside an acid-proofed concrete bund to ensure that all spillage is captured and kept separate from other process streams.

After acid wash, the carbon is pumped to an elution circuit that includes elution columns, a strip solution tank, a strip solution pump, and a strip solution heat exchanger. The elution circuit operates in closed circuit with electro-winning cells.

Strip solution heat exchangers maintain the strip solution at 145 °C during the stripping cycle and ensure that the temperature of solution entering the electro-winning cells is below 100 °C.

Eluate flows directly from the top of the elution column to a loaded solution tank after cooling through heat exchangers. The eluate is pumped from the loaded solution tank to electro-winning cells to recover gold and silver as sludge. Barren solution from electro-winning gravitates back to the strip solution tank. The sludge is drained from the electrowinning cells and vacuum filtered before refining.

17.3.5 Refining

Filtered cake from electro-winning is dried in two drying ovens and directly smelted with fluxes in two induction furnaces. Gold-silver doré is poured into doré moulds. Gold-silver doré bars are weighed, stamped, sampled and stored in a safe ready for dispatch.

Furnace exhaust is passed through a wet scrubber to remove any entrained particles and then vented through a stack.

17.3.6 Detoxification

Tails from the CIL are thickened and fed to a detox reactor at 45% solids w/w. Cyanide destruction is carried out using the SO2/Air process using sodium metabisulphite. Slurry produced from the detoxification stage is pumped to the final tailings thickener.

An HCN detector will monitor for airborne gas and a cyanide analyzer will be used to monitor cyanide levels and ensure that target cyanide levels are achieved.

17.3.7 Tailings thickener

The final tailings thickener combines tailings streams from flotation and detoxification. Thickener overflow is recirculated to the process water system. Thickener underflow is pumped to the tailings storage facility at 50% solids.

17.4 Reagents and Power Consumption

Reagents are prepared in a separate contained area and are bunded to control any spillage. Tank storage capacity is based on reagent consumption rates to supply the process without any interruption.

A summary of the estimated reagent consumption rates is provided in Table 17-2.

Electrical power for processing is estimated at 9 MW.

Table 17-2 Reagents and Consumables Summary

Reagent	Consumption kg/t Mill Feed
Grinding Media	0.394
Potassium amyl xanthate (PAX)	0.065
Methyl isobutyl carbinol (MIBC)	0.050
Lime	0.055
Test Reagent	0.000
Carboxymethylcellulose (CMC)	0.065
Sodium Cyanide	0.100
Copper Sulphate	0.0014
SMBS	0.022
Sodium Hydroxide	0.002
Hydrochloric Acid	0.0003
Leach-Aid	0.0001
Flocculant	0.001
Flux	0.001
SiO2	30%
Borax	40%
Niter	10%
Soda Ash	20%
Carbon	0.050

17.5 Process Water and Power

The raw water supply to the process plant is described in Item 18 (Infrastructure), along with fire water and potable water.

Raw water is pumped to a Fresh Water storage tank with 14 m diameter and 18 m height. Make up water and fire water for the plant is drawn from the Fresh Water Tank. Water recycled in the plant area is pumped to a Process Water tank with 18 m diameter and 18 m height.

A detailed plant water balance has not yet been completed.

Site power, estimated to be 10.5 MW, is described in Section 18; the process facilities are estimated to use up to 9 MW for operations.

17.6 Assay and Metallurgical Laboratory

The assay laboratory will be equipped with the necessary analytical instruments to provide all routine assays for the mine, the concentrator, and the environment departments. The most important of these instruments includes:

- sample preparation equipment,
- fire assay equipment,
- atomic absorption spectrophotometer (AAS),
- and Leco furnace.

The metallurgical laboratory will undertake all necessary test work to monitor metallurgical performance and, more importantly, to improve process flowsheet unit operations and efficiencies. The laboratory will be equipped with laboratory crushers, ball and stirred mills, particle size analysis sieves, flotation cells, filtering devices, balances, and pH meters.

18.0 Project Infrastructure

The following Section discusses the project infrastructure including the on-site infrastructure, off-site infrastructure (external power supply transmission line), the Tailings Storage Facility (TSF) and water management system.

18.1 On-Site Infrastructure

On-site infrastructure includes:

- Electrical Substation
- Tailing Storage Facility
- Water Storage Pond
- Maintenance and Truck Shop
- Admin/Dry Building
- Assay Laboratory
- Cold Storage Warehouse
- Access roads
- Water Supply
- Wastewater treatment systems
- Solid waste disposal facilities and sewage plant
- Communication systems
- Medical facilities
- Site support systems including workshops, maintenance shop, warehousing and security

A high level layout of the on-site infrastructure is shown in Figure 1-3.

18.1.1 Power and Electrical Distribution

Electrical costs are factored from other recent similar studies. Offsite supply is described in Section 18.6, and on site distribution described in Section 18.3.1. Site power will be distributed to various modular electrical rooms on site by means of an overhead line to the following areas:

- primary crushing
- tailings/water management
- explosive manufacturing
- Maintenance/truck shop

18.1.2 Maintenance and Truck Shop

The truck shop building will be a pre-engineered steel building with insulated roof and walls. The building will be supported on concrete spread footings with concrete grade walls along its perimeter.

The building will house a wash bay complete with pressure water, three repair bays, warehouse area, warehouse/parts storage, welding area, machine shop, emergency vehicle parking, first aid room, electrical room, mechanical room, compressor room and a lube storage room. The warehouse and repair bays will be serviced by two overhead cranes.

The wash bay will include a cattle guard and truck washings will be collected in a sump and pumped to the process plant.

18.1.3 Access Road

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 existing Spanish Mountain 1300 Forest Service Road (FSR). This road currently travels through the proposed mine site; it will require rerouting in order to accommodate the location of the north waste dump and open pit. Access to this FSR route through the site will be maintained throughout the LOM.

18.1.4 On-site Roads

On-site roads are differentiated from haul roads in that they are defined as access roads to all facilities including the TSF, and for maintenance traffic between the two mine pit locations. These roads are unpaved and provide service/maintenance access for vehicles to all areas of the proposed facilities.

The on-site service roads will join at strategic points, to the main access road and cross various haul roads at specific points of the mine haulage route.

The haul roads between the pit, the primary crusher at the plant site, the RSF, and the TSF will be constructed with a top course of crushed mine rock.

18.1.5 Waste and Sewage Systems

Wastewater or sewage generated on-site will be treated at the sewage treatment plant. Treated effluent generated at the sewage treatment plant will be compliant with local and national regulations.

All potable water that is generated and consumed on-site for domestic use is expected to report to the sewage treatment plant for treatment prior to discharging to the environment. Therefore, it is assumed that the volume of sewage generated is equivalent in volume to the amount of potable water produced on site for domestic use.

18.1.6 Communication System

A satellite-based system will be needed for external voice and data communications services. An on-site network will be established that will connect buildings and radio transceivers will be used for remote monitoring and control. An ultra-high frequency (UHF) radio system will be used for mobile communication.

18.1.7 Administration/Dry Building

The Administration and Dry Building will be a modular building supported on concrete spread footings, complete with furniture and equipment.

18.1.8 Assay Laboratory

The assay laboratory will be a pre-fabricated modular structure located close to the mill building. The building will house all necessary equipment for metallurgical grade testing and control.

18.1.9 Cold Storage Warehouse

The Cold Storage warehouse will be a pre-engineered sprung steel structure with an un-insulated fabric cover. The building will be supported on pre-cast concrete lock blocks on a prepared gravel surface.

18.1.10 On-Site Explosives Manufacturing and Storage

Contractor blasting services will utilize an on-site explosives storage and mixing area, which will consist of fenced off storage tanks and containers, Ammonium Nitrate and Emulsion silos, an office trailer, a truck shed and shop, a compressor and a generator. A separate facility for storage of detonators will be required.

18.2 Building Services

All process areas will be heated to a minimum temperature of 5^0C during the cold season, by providing propane-fired heating units along perimeter walls and above doorways.

All staff-occupied areas will be heated to a minimum of 20 ^{0}C during the cold season, by supplying filtered and tempered outdoor air mixed with return air. The air will be distributed through ductwork.

Plumbing, fire protection and dust control will be provided as per national codes and accepted industry practices.

18.3 Site Utilities and Support System and Support Systems

18.3.1 Electrical Substations and Power distribution

The primary distribution switchgear will be located inside the main substation area. The total estimated running load for all site facilities is approximately 10.5 MW. Secondary system voltages utilized will include for major drives and secondary distribution, motor control centers, long-line piping heat tracing, and lower voltages for lighting, instrumentation, controls and general usage.

Electrical substations next to the plant will be fed by overhead lines and insulated cables via duct banks. Pipe racks will also be used where possible for major cable tray routes within the plant area. Cable trays that are at grade level and exposed will have hi-visibility covers for awareness and mechanical protection. The line will also service the primary crushing and mining facilities, as well as a line that will service water supply stations, tailing, the explosives plant and the waste management facility.

Pre-fabricated and pre-assembled E-Houses will be utilized to house all electrical distribution equipment.

18.3.2 Fuels Storage and Distribution

The primary project diesel fuel storage will be in two bulk storage tanks located near the truck shop complex. Fuel dispensing facilities, including light vehicle as well as fast-fill facilities for mining equipment, will be included.

18.3.3 Potable Water Supply, Storage and Distribution

Potable water will be required to meet demands for drinking, food preparation, clean-up in kitchen and dining facilities, personal hygiene (toilets and/or urinals, sinks and showers), laundry, and for safety shower/eye wash stations.

Fresh water will be treated in the Potable Water Treatment Plant to meet the criteria of local and national water quality regulations and guidelines.

18.3.4 Water Treatment Plant

Water will be reclaimed for use within the process plant and excess water will be pumped to a water treatment plant prior to discharging to the environment.

18.4 Tailing and Water Management

18.4.1 Design Basis and Operating Criteria

The principal objective of the TSF is to provide secure containment of all tailings solids and PAG/ML waste.

The metallurgical process involves a rougher flotation circuit to produce rougher tailings. The rougher concentrate is cleaned in two stages and the combined cleaner/recleaner tails are scavenged by gravity concentration to produce the final cleaner tailings stream which is combined with the concentrate that has been reground and subjected to carbon in leach and cyanide detoxification, and which is assumed to be PAG and ML if allowed to oxidize. The rougher and cleaner/concentrate tailings streams will be transported from the plant site to the TSF in separate pipelines. Each tailings stream will be deposited independently; the rougher tailings will be discharged along the TSF embankments to create tailings beaches, and the cleaner tailings will be discharged sub-aqueously in the supernatant pond and progressively encapsulated by the rougher tailings.

The TSF capacity at all stages of the mine life includes the supernatant pond volume and allowances for wave run-up, post-seismic settlement, sloping beaches and containment of the inflow design flood.

18.4.2 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 include filter and transition zones to ensure proper filter relationships between adjacent zones, and to convey drainage within the embankment. A downstream shell zone comprises the majority of the embankment materials.

The starter dam, 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, and associated freeboard allowances. The TSF embankments will be constructed in stages; each stage will provide the required capacity for the period until the next stage of construction is completed. The final capacity of the TSF will be approximately 39 Mt of tailings, 25 Mt of PAG/ML waste rock, plus the supernatant pond volume and freeboard allowances.

The starter embankments will be constructed with 2.25:1 upstream and downstream slopes. The embankments will be progressively expanded using centerline construction methods while maintaining a 2.25:1 downstream slope.

18.4.3 Construction Materials

The TSF embankments will be constructed using suitable waste rock and overburden (low permeability glacial till) from the open pit.

18.4.4 Tailings Distribution Systems

The rougher tailings will be discharged into the TSF from a series of large diameter valved off-takes located along the embankments to facilitate tailings beach development.

The cleaner tailings will be discharged separately to allow subaqueous deposition and for progressive encapsulation by the rougher tailings.

18.4.5 Reclaim System

Reclaim water for use in the mill processes will be pumped from a floating barge on the TSF to the water management pond. The barge will be positioned at the north end of the TSF to minimize pumping distance to the water management pond. The reclaim water will subsequently be pumped from a secondary floating barge on the water management pond to a process water tank located outside of the mill building. The tank will store a 24-hour supply of mill process water.

18.4.6 Water Management

The water management pond will serve as a primary site water management facility, providing a buffering stage for process water, direct precipitation and runoff to be held prior to treatment and discharge offsite.

Surface diversion ditches will capture and divert non-contact water around the TSF for direct release to the environment. 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 directed through an overflow channel to Winkley Creek and, eventually, to Quesnel Lake.

Seepage collection ponds 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.

18.5 Waste Rock Management

18.5.1 Waste Rock Production

MMTS developed a 10,000 t/d production schedule which defined the amount of mineralized material, waste rock, overburden and undefined material produced annually over the mine life. MMTS also identified and categorized the waste rock based on its geochemical characterization (see Section 16.9).

18.5.2 Waste Disposal Strategy

Suitable waste rock and overburden will be hauled from the open pit to the TSF embankments for use as construction materials. The PAG/ML waste rock will be deposited within the TSF in such a manner that it is progressively encapsulated by the tailings and saturated by the supernatant pond.

18.6 Off-Site Infrastructure

The Project requires 10.5 MW of peak load for 10,000 t/d operation demand. The power will be supplied by a new transmission line interconnecting the SMG site to BC Hydro's power system.

In the previous 2012 "Technical Report and Preliminary Economic Assessment of the Spanish Mountain Gold Project, Likely, BC" report Stantec previously performed an assessment for an interconnection transmission line to BC Hydro's power system for the Project. This assessment did not reflect BC Hydro's system impact study (SIS), which is yet to be completed. The result of BC Hydro's SIS will determine the most suitable point of interconnection to BC Hydro's grid, and the estimated costs associated with system reinforcement.

The previous report stated the following:

"According to the latest preliminary results from BC Hydro's SIS and considering the constraints due to land property issues for expansion at the existing BC Hydro Soda Creek substation, BC Hydro confirmed a new 230 kV transmission line directly from a new BC Hydro 230 kV switching station adjacent to BC Hydro's existing 500 kV McLeese Capacitor station to the SMG site is the only technically leading option for power supply".

This option has been adopted for current PEA.

19.0 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. 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.

Since 2016 the price of gold has fluctuated between US$1,060 and US$1,550 per ounce. The 1, 2, 5, 8 and 10 year trailing average gold prices are all above US$1,300 per ounce. A gold price of US$1,275 per ounce, and a silver price of US$18 per ounce, are considered by the QP's as reasonable with respect to the current market and have been used for this Preliminary Economic Assessment.

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.

20.0 Environmental Studies, Permitting and Social or Community Impact

20.1 Environmental Studies

Environmental studies—including studies on surface and groundwater quality and quantity, geochemistry, climatology, fish and fish habitat, wildlife, and vegetation—were initiated in 2007 at the Project site and continued through 2011.

Water quality monitoring sites were established throughout the Project area to characterize existing water quality conditions. Water quality samples from within the claim boundary have consistently shown concentrations of total and dissolved metals that exceed limits set by the Canadian Council of Ministers of the Environment (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.

Site-specific fish and fish habitat assessments confirmed the presence of rainbow trout in Spanish Creek, Cedar Creek, Nina Lake, Boswell Creek, Boswell Lake, and Winkley Creek. Chinook salmon, dace, and burbot were captured near the mouth of Cedar Creek; juvenile Chinook were captured, and adult Coho salmon were detected near the mouth of Spanish Creek.

The Wells Grey subpopulation of mountain caribou is located outside of the Project area in the upper catchment of Black Bear Creek, approximately 15 km to the northeast of the Project. The range of the Quesnel Lake North population of grizzly bear covers the Project area. Other flora and fauna species in the Project area are typical for the region.

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

20.2 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.

Static and kinetic geochemical tests of tailings and waste rock were performed to assess the metal leaching and acid rock drainage (ML/ARD) for the materials to be managed at the Spanish Mountain project. Static tests included acid-base accounting (ABA), composition analyses by aqua regia dissolution followed by ICP-MS analyses, and mineralogical evaluation. Kinetic tests included laboratory humidity cells and field barrels to assess sulphate oxidation rates and metal leaching potential. The Spanish Mountain resource is indicated to have a low potential for ML or ARD, especially if waste segregation strategies can be incorporated into proposed mining methods.

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.

The federal *Fisheries Act* prohibits the serious harm of fish without specific authorization. Construction of the tailings storage facility in the Cedar Creek basin may require a Schedule 2 Amendment under the Metal Mining Effluent Regulations (MMER) of the *Fisheries Act*. The MMER were developed to control the deposit of mine tailings and waste matter into fish-bearing waters. Fisheries and Oceans Canada (DFO), Environment and Climate Change Canada (ECC), and Natural Resources Canada (NRCan) will conduct a thorough analysis of tailings management options, which includes public consultation, to ensure that the proposed use of the waterbody is the most appropriate option, and a comprehensive fish habitat compensation plan will be required to ensure no net loss of fish habitat. Fish habitat compensation will also be required to balance any loss of fish habitat in Spanish Creek as a result of pit development or waste rock placement, and in Cedar Creek as a result of reduced flows from diversion of surface runoff around the TSF. Monitoring will be carried out during the life of the Project, including its post-closure phase, to ensure efficacy of the water quantity and quality controls as they affect fish habitat.

20.3 Permitting

The Environmental Assessment 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 CEAA. Detailed environmental and socio-economic baseline studies were then initiated and conducted over a two year period. Future updates to these studies, along with any completed feasibility studies, will form the basis of an impact assessment, which will be submitted as part of the EA and reviewed by regulators, First Nations, and the public.

Advancement on the EA was halted by Spanish Mountain Gold while project design updates were completed, however the provincial EA process remain in progress. SMG has committed to keeping the process active through the provision of quarterly updates to the EAO, until such time as work resumes on the document preparation.

In November 2018, the BC Government passed a new Environmental Assessment Act. SMG will have six months from the time the new Act comes into force (expected late 2019) to file a notice indicating a preference to remain in the grandfathered Act and complete the EA process within three years, or to transition to the new Act. Under either option, SMG will have the opportunity to work with the Environmental Assessment Office to utilize the work completed in previous years as much as possible to move the process forward.

In August 2019, the new federal Impact Assessment Act came into force, replacing the Canadian Environmental Assessment Act, 2012. As a result, the ongoing comprehensive study initiated by SMG in 2011, which was being conducted under the former Canadian Environmental Assessment Act, was terminated per the transitional provisions of the Impact Assessment Act. SMG will be required to re-initiate the federal EA under the new Act in order to gain federal permits.

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. The principal required provincial permits are expected to be a *Mines Act* permit and an *Environmental Management Act* permit. Federally, the principal permits are expected to be an *Explosives Act* permit, and authorization under the *Fisheries Act*.

20.4 Social or Community Requirements

Public comment in relation to the Project must be sought, addressed, and documented through public open houses, meetings and presentations, and through the provincial and federal online EA registries.

The Project is located 6 km east of the community of Likely, BC, which has a population of approximately 350 people. Williams Lake is located 66 km southwest of the Project and has a population of approximately 11,000. Quesnel is located 90 km northwest of the Project and has a population of approximately 10,000 inhabitants. Other communities in the area include Horsefly, Black Creek, Keithley Creek, Quesnel Forks, and Big Lake.

The Project is situated within the asserted traditional territories of the T'exelc (Williams Lake) and Xats'ull/Cmetem' (Soda Creek) First Nations, both of whom are member nations of the Northern Secwepemc te Qelmucw (Northern Shuswap Tribal Society Council), as well as the Lhtako Dene Nation (Red Bluff Indian Band), which is part of the Carrier Chilcotin Tribal Council. SMG has signed cooperation agreements with each of the three First Nations. These agreements, among other things, establish a timeline for negotiating a participation agreement as a part of the overall environmental assessment and permitting process. As the Environmental Assessment is not currently being advanced, the timelines set in the cooperation agreements have not been met. The parties are expected to revisit and renew the agreements once Feasibility level design is advanced to support recommencement of the Environmental Assessment process.

An updated Archaeological Impact Assessment was conducted in 2019. It confirmed that there were unlikely to be impacts to heritage resources.

Community and First Nations consultation has been initiated by SMG and will continue throughout the development of the Project. Traditional Knowledge and Land Use studies have been completed with the T'exelc and Xats'ull/Cmetem' First Nations.

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. 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 cost is indicated in Section 21.0.

21.0 Capital and Operating Costs

21.1 Capital Cost Estimate

The total estimated pre-production capital cost for the design, construction, installation, and commissioning for all facilities and equipment for the Spanish Mountain Gold Project is shown in Table 21-1 below.

The accuracy of the estimate is ±40%. This study has been prepared with a base date of Q4 2019 with no provision for escalation. All Capital and Operating costs are reported in Canadian dollars unless specified otherwise; an exchange rate of US$0.75 to C$1.00 has been used for any conversions.

Capital cost estimates have been developed by:

- MMTS – Mining, Process, General Site, On-site and Off-site Infrastructure
- KP – Material take-offs for the TSF (including the associated tailing delivery and return pipelines to and from the process plant), Water Management and Water Treatment.
- SMG – Environmental, Owner's Costs.

MMTS is responsible for the assembly of the overall estimate.

Initial capital has been designated as all capital expenditures required prior to mill start-up for producing doré for shipment to buyers.

The estimate covers the direct field costs of executing this project, plus the indirect costs associated with design, procurement, and construction efforts.

21.1.1 Sustaining Capital Cost Estimate

Sustaining Capital includes replacement equipment purchases, tailing dam construction, water treatment operations, and continued open pit mining development. Any work which is scheduled to begin after plant start-up is generally included in the sustaining capital costs.

Table 21-1 Capital Cost Summary

Direct Costs	**Initial Capital Cost (M$)**
Overall Site	6.7
Open Pit Mining	70.2
Ore Handling	24.0
Processing Plant	53.4
Tailing Storage Facility & Water Management	46.7
Environmental	12.0
On-Site Infrastructure	24.0
Off-Site Infrastructure	17.1
Sub-Total	**254.1**
Indirect Costs	
Project Indirects	58.9
Owner's Costs	9.3
Contingencies	41.5
Sub-Total	**109.7**
Total Initial Capital Cost	**363.8**

Table 21-2 Initial and Sustaining Capital Cost Estimates (M$)

Year	LOM	Initial	Sustaining	-2	-1	1	2	3	4	5	Y6-11	Y11-15	Y16-20
DIRECT COSTS:													
Site		$6.7	$0.0	$2.7	$4.0								
Mining		$70.2	$32.2	$36.2	$34.0	$18.8	$0.1	$0.0	$0.3	$0.2	$12.3	$0.5	$0.0
Processing		$77.5	$0.0	$31.0	$46.5								
Tailings Dam and Water Management		$46.7	$25.6	$18.7	$28.0	$9.8	$1.5	$1.4	$1.4	$3.1	$7.0	$1.4	$0.0
Environmental		$12.0	$0.0	$4.8	$7.2								
Site Infrastructure		$41.1	$0.0	$16.4	$24.7								
Total Direct Costs	**$311.8**	**$254.1**	**$57.8**	**$109.8**	**$144.3**	**$28.6**	**$1.6**	**$1.4**	**$1.7**	**$3.3**	**$19.3**	**$1.9**	**$0.0**
INDIRECT COSTS:													
Indirect Costs		$58.9	$0.0	$23.6	$35.3								
Owners Costs		$9.3	$0.0	$3.7	$5.6								
Reclamation Costs		$0.0	$45.0						$1.0	$1.0	$5.9	$24.9	$12.2
Salvage Values		$0.0	-$15.0									-$10.0	-$5.0
Total Indirect Costs	**$98.2**	**$68.2**	**$30.0**	**$27.3**	**$40.9**	**$0.0**	**$0.0**	**$0.0**	**$1.0**	**$1.0**	**$5.9**	**$14.9**	**$7.2**
CONTINGENCY COSTS:													
Contingency		$41.5	$0.0	$16.6	$24.9								
Total Contingency	**$41.5**	**$41.5**	**$0.0**	**$16.6**	**$24.9**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**
TOTAL COSTS:													
Total Capital		$363.8	$87.8	$153.6	$210.1	$28.6	$1.6	$1.4	$2.7	$4.3	$25.2	$16.8	$7.2
Total Project Capital	**$451.5**	**$363.8**	**$87.8**	**$153.6**	**$210.1**	**$28.6**	**$1.6**	**$1.4**	**$2.7**	**$4.3**	**$25.2**	**$16.8**	**$7.2**

21.1.2 Site Development, On-site and Off-site Infrastructure

An initial CAPEX was completed for site development, on-site and off-site infrastructure. Section 18 describes the components included in this Estimate.

- Site Preparation
  - Access Roads
  - On-Site Roads
- Power and Electrical Distribution
- Site Controls and Communication
- Ancillary Buildings
  - Maintenance and Truck Shop
  - Administration/Dry Building
  - Assay Laboratory
  - Cold Storage Warehouse
  - Medical Facilities
- Site Services and Utilities
  - Water Supply
  - Wastewater Treatment
  - Solid waste disposal facilities and sewage plant
  - Fuel Storage and Distribution
- Plant Mobile Equipment
- Transmission Line and Substations
- EPCM

Capital cost estimates for these components were detailed out as part of the 2012 Spanish Mountain Gold study (Tetra Tech, 2012), and updated for this PEA.

21.1.3 Mining

An initial and sustaining CAPEX was completed for the following mining components:

- Pre-production cost is estimated for 7.7 Mt of waste and 1.0 Mt of mineralized material mined in pre-production. The costs include hauling 2.5 Mt of suitable waste rock to the tailing's embankment for construction. Mine development will be initially undertaken by a contractor and will consist primarily of haul road construction and upper bench drilling and blasting. It is anticipated that the Owner's mine equipment fleet will be available for all mining activities thereafter.
- Initial mine equipment includes the total fleet requirement to meet the total material production in the pre-production. Sustaining capital includes total fleet requirements to meet the material production scheduled, as well as an estimated replacement schedule based on equipment usage.

- The equipment pricing is based on new units delivered to the mine, with all transportation, assembly and commissioning costs included. Most unit prices 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.
- Pit dewatering and depressurization costs are estimated. It includes drilling vertical wells and horizontal holes, pump installations and maintenance.
- 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.
- Site-preparation cost contains an allowance for clearing and grubbing, drainage ditches, topsoil removal and acquiring granular materials for road surfacing.
- Road construction costs are estimated for approximately 11 km of haul roads to be undertaken by a contractor.
- Salaries and costs for mine and engineering staff, and consultants during the pre-production period are included.
- Other capital cost allowances for GPS Guidance on equipment, computer supplies, mine rescue gear, surveying equipment, and communications facilities are included.
- The cost of blasting facilities has been estimated based on vendor recommendations for the site.
- Three percent of the mobile equipment fleet capital is included for spare parts such as truck tires, loading buckets, shovel teeth, drill bits, etc. Due to the proximity of the mine to other operating mines and service centres, it is anticipated that this amount carried at the mine site will be sufficient.

21.1.4 **Processing**

An initial CAPEX was completed for the process plant, which includes the following components:

- Crushing
- Grinding
- Flotation
- Gravity Concentration
- Cyanidation (CIL)
- Reagents and Consumables
- Plant Services
- Tailings Thickening
- Effluent Treatment
- Capital Spares
- First Fills
- Temporary Construction
- EPCM

Costs are estimated by MMTS based on a combination of benchmarking recently constructed projects in North America and a factored and inflated estimate of the 40,000 t/d process plant capital cost estimate from the 2012 study for Spanish Mountain Gold (Tetra Tech, 2012).

Sustaining costs for the process plant are assumed to be covered in the process plant operating costs.

21.1.5 **Tailings Storage Facility and Water Management**

An initial and sustaining CAPEX is completed for the following components of waste and water management:

- Contractor mobilization and demobilization.
- Site preparation for the TSF embankment footprints, laydown areas, topsoil and unsuitable stockpiles including clearing and grubbing, wetland dewatering and excavation, select service road construction, placement of a wearing course on the laydown area, construction dewatering and sediment and erosion control Best Management Practices (BMPs).
- Earthworks costs for both TSF embankments. The total earthworks costs are integrated between mine operating costs and the tailing and water management costs, with mine operating costs covering most of the material haulage costs from the open pit and borrow source.
- Diversions ditches, Boswell Lake diversion embankment and overflow channel.
- Sediment control ditches for waste rock and ore stockpiles.
- Water management pond construction.
- Tailings distribution and embankment seepage collection and recycle systems.
- TSF embankment monitoring instrumentation (piezometers and inclinometers).
- Sustaining capital covers the operating costs of the water treatment plant. These costs have been estimated beyond the operation of the open pit and mill.
- EPCM.
- Indirects.

Development of initial and sustaining capital costs for the waste and water management facilities necessitated assumptions of the geotechnical site conditions which must be verified. The cost estimate is compiled using information from similar projects, engineering experience and unit rates built up using first principles, based on standard contractor rates in BC.

21.1.6 **Environmental**

Habitat compensation costs for the TSF are developed assuming that any fish habitat lost or altered as a result of mine development will be replaced, in accordance with DFO policy. Exact habitat compensation requirements will need to be determined with DFO as part of future permitting exercises. Both direct footprint impacts and indirect downstream flow reductions are considered potential harmful alteration in the habitat compensation assessment. Instream compensation areas are calculated based on an estimated mean channel width of 5 m for fish-bearing mainstem channels and 3 m for fish-bearing tributary channels; riparian compensation areas are calculated based on 30 m setback widths for mainstem channels and 15 m setback widths for tributaries. The compensation areas of mainstem channels downstream of the TSF that would be harmfully altered due to reduced flows are also calculated based on an estimated mean channel width of 5 m.

Development of the TSF will directly affect Nina Lake, the mainstem of Cedar Creek and several unnamed tributaries. It will also indirectly affect the lower reaches of Cedar Creek. Fish habitat compensation ratios are calculated as 2:1, with assumed unit area capital costs of $150,000/ha for instream habitat and $50,000/ha for riparian habitat. Based on these unit area costs, fisheries compensation is anticipated to cost approximately $10 million, included as initial CAPEX; $1 million per year has also been allocated for environmental monitoring, which has been capitalized during pre-production and included in G&A operating costs throughout the remainder of the project.

21.1.7 **Indirect Costs**

The Project indirect costs include:

- construction: temporary works (lighting, water supply, sewage, power), craneage, equipment rentals, garbage and hazardous waste disposal, quality assurance, surveying, medical/first aid, mobilization/demobilization, warehousing, laydown areas, personnel transportation, safety, security)
- spares: capital/commissioning
- initial fills: one-month supply of ball grinding media, mill liners (not included), reagents, fuel, lubricants, mining supplies allowance
- freight and logistics: land and ocean transportation, loading and offloading, including craneage, marshalling yard, ocean transportation, customs duties and brokerage
- commissioning and start-up costs
- EPCM allowance: based on percentages of the direct costs
- vendors’ assistance

Indirect items such as exploration, land acquisition, royalty buyouts, future studies, and permitting costs are excluded from the capital estimate for the Project.

Working capital has not been included in the capital cost estimate.

21.1.8 **Owner’s Costs**

The Owner’s costs are estimated to be $9.3 million. Owner’s costs are abated by the assumptions that the head office will absorb some the costs and they will not be distributed directly to the project. The costs distributed to the project include:

- Builder’s Risk Insurance
- Construction Management
- Accounting
- Procurement and Warehousing
- Safety and First Aid
- Administration
- Facilities Services

- Site Maintenance
- Facilities Furniture
- Safety Supplies and Equipment
- Telephone and Communication Supplies and Equipment
- Office Supplies and Equipment
- Medical Services and Supplies
- Local Permitting
- Local Recruitment
- Systems and General Training
- Warehousing
- Housing Costs
- Travel Allowances
- Reclamation Bonding (based the sustaining CAPEX described in the next Section).

21.1.9 Reclamation and Salvage Values

Sustaining CAPEX of $42.9 million is estimated for reclamation activities and is offset by estimated salvage values of $15.0 million for mobile equipment and facilities that have been decommissioned.

Reclamation cost estimates are based on the following estimated unit rates.

Table 21-3 Reclamation Unit Costs

Item	Unit Cost
WRSF Tops ($/ha)	$30,000
WRSF and Tailings Faces ($/ha)	$60,000
Tailings Tops ($/ha)	$45,000
Roads and Berms ($/ha)	$10,000
Infrastructure ($/ha)	$20,000

Progressive reclamation is planned between Year 4 and Year 16. Salvage values are split up and applied at the end of pit mining in Year 12 ($10.0 million), and at the end of reclamation activities in Year 16 ($5.0 million).

21.1.10 Contingency

The overall contingency for the Project is $41.5 million.

The estimated contingencies are for undefined items of work which are incurred within the defined scope of work covered by the estimate, which cannot be explicitly foreseen or described at the time the estimate was compiled, due to a lack of complete accurate and detailed information. Therefore, the contingency is an integral part of the estimate. The contingency is not to be considered as a compensating factor for estimating inaccuracy, nor is it intended to cover such items as any potential "changes in project scope", "Acts of God", prolonged labour strikes, labour disruptions beyond the control of the project manager, currency fluctuations or cost escalation beyond the estimated rates.

It is considered that this estimate will adequately cover minor changes to the current scope, to be expected during the next phase of the Project.

No provision is made, or contingency allowed, for major design amendments or changes to the scope, which may result from additional test work or pilot plant testing which would be carried out to verify the current design in the next phase of the project. No provision is made, or contingency allowed, for major design amendments or changes to the scope, which may result from additional geotechnical studies or further investigation of the site conditions.

21.2 Operating Cost Estimate

Operating costs for the project are broken down in the following three categories:

- Mining
- Process
- Tailings
- General and Administration (G&A)

Table 21-4 Unit Operating Costs

Area	Unit Cost
Mining ($/t mined)	$2.48
Mining ($/t milled)	$10.73
Processing ($/t milled)	$6.14
Tailings ($/t milled)	$0.16
G&A ($/t milled)	$2.06
Total ($/t milled)	**$19.10**

A summary of the life of mine cash operating and all-in sustaining cost/oz. is set out in the Table below (in Canadian funds).

Table 21-5 Life of Mine Cash Operating and All-in Sustaining Costs/oz.

Unit Production Costs per ounce	First 5-Yrs	Life of Mine
Cash Cost	$616	$657
All-in-Sustaining Cost (AISC)	$692	$733
Total Cost	$1,035	$1,075

In addition to cash operating costs, all-in sustaining costs include sustaining capital, refining charges and royalties. Total Costs include initial capital and reclamation costs.

Costs for mining and processing have been built up using first principles, using the following fuel and power cost inputs.

Table 21-6 Fuel and Power Input Costs

Item	Unit Cost
Fuel ($/L)	$1.00
Power ($/kWh)	$0.065

21.2.1 Mine Operating Costs

Mine operating costs are built up from first principles, based on the following breakdown by area of mine operation in Table 21-7.

Table 21-7 Mine Operating Cost Breakdown

Area of Mine Operation	$/t Mined	$/t Milled
Drilling	$0.23	$0.98
Blasting	$0.31	$1.35
Loading	$0.29	$1.26
Hauling	$0.90	$3.87
Pit Support	$0.43	$1.85
Geotechnical	$0.05	$0.21
DIRECT COSTS - Subtotals	$2.20	$9.51
Mine Operation/Maintenance GME	$0.18	$0.77
Mine Engineering GME	$0.10	$0.45
TOTAL GME COSTS	$0.28	$1.22
Total Operating Cost	**$2.48**	**$10.73**

The largest component of these operating costs is hauling. Haul cost estimates are based on simulated haul cycles to the crusher, stockpiles, WRSF's, and tailings dam. These simulated cycles times are applied over the scheduled tonnes for each period of mine operations. During open pit operations, average annual hauler productivities range from 200 to 375 tonnes/operating hour.

21.2.2 Process Operating Costs

Process operating costs are built up from first principles. Table 21-8 summarizes the process operating costs.

Table 21-8 Process Operating Cost Breakdown

Area	$/t milled
Labour	$1.85
Consumables	$2.48
Power	$1.40
Maintenance	$0.41
Total	**$6.14**

21.2.3 Tailings Operating Costs

Tailings operating costs of $0.16/t milled have been added to cover operating costs of the tailings distribution and embankment seepage collection and water reclaim systems.

21.2.4 General and Administration Operating Costs

General and Administration (G&A) operating costs are built up from first principles. The total annual cost for G&A items is estimated to be $7.5 million, or $2.06/t milled. Head office expenses are assumed to be separate and exclusive from the project specific General and Administration costs outlined below.
G&A costs include all salaried and hourly labour not assigned to mine or process operations. This includes:

- General Management
- Administration
- Human Resources
- Reception
- Health, Safety and Environmental
- Security
- Procurement and Warehousing
- Accounting
- IT
- Janitorial
- Site Services

It also includes consumables and contractors not covered under the mine and process operations. This includes:

- Office Supplies and Stationary
- Professional Associations and Publications
- Insurance
- Travel
- Site Communications
- Computer and IT Services
- Site Services: potable water, sewage, HVAC, garbage, etc.
- Community Relations
- Recruitment
- Training
- Site Power
- Protective Equipment and Training Supplies
- Medical Services and First Aid Supplies
- Security Supplies
- Environmental Equipment, Supplies and Monitoring
- Purchasing and Logistics / Warehouse costs

- External Assays and Testing
- Janitorial
- Light Vehicles
- Powerline Maintenance
- Road Maintenance
- Crew Transportation

22.0 Economic Analysis

22.1 Cautionary Statement

The results of the economic analyses discussed in this section represent forward- looking information as defined under Canadian securities law. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented 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 Q4 2019 Canadian dollars, unless specified otherwise. The economic analysis is run over the entire project life, comprising two years of construction and 11 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, 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 Q4 2019, has been used to estimate federal, provincial, and other taxes. Some additional details are included in Section 22.3.

The preliminary economic assessment is based on resources, not reserves. Resources are considered too speculative geologically to have economic considerations applied to them, so the project does not yet have proven economic viability.

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 21.0.

Table 22-1 Inputs for Economic Analysis

Parameter	Value	Units
Gold Price	$1,275	US$/oz
Silver Price	$18	US$/oz
Currency Exchange Rate	0.75	C$:US$
Gold Payable	99.8%	
Silver Payable	90.0%	
Gold Refining Terms	$1	$/oz
Silver Refining Terms	$0.6	$/oz
Doré Transport Costs	$1	$/oz
Doré Insurance Costs*	0.15%	
Royalty**	1.5%	
Mining Cost***	$2.48	$/t mined
Gold Process Recovery	91%	
Silver Process Recovery	27%	
Processing Costs	$6.14	$/t milled
General & Administration Costs	$2.06	$/t milled
TMF Operating Costs	$0.16	$/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 0, 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 $2.48/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	39.1	Mt
Au Grade	1.00	g/t
Au Produced	1,145	koz
Ag Grade	0.74	g/t
Ag Produced	250	koz
Waste Mined	138.5	Mt
Strip Ratio	3.5	t:t
Initial Capital	364	$M
Sustaining Capital	58	$M
Cash Costs (LOM)*	511	US$/oz
AISC (LOM)**	549	US$/oz
Total Costs (LOM)***	807	US$/oz
Net Cash Flow (Pre-Tax)	716	$M
Pre-Tax (SMG)		
NPV, 5%	414	$M
IRR	23%	%
Payback	3.5	Years
Post-Tax (SMG)		
NPV, 5%	325	$M
IRR	21%	%
Payback	3.5	Years

** Cash Costs include all operating costs associated with the production and sale of gold.*
*** 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.*

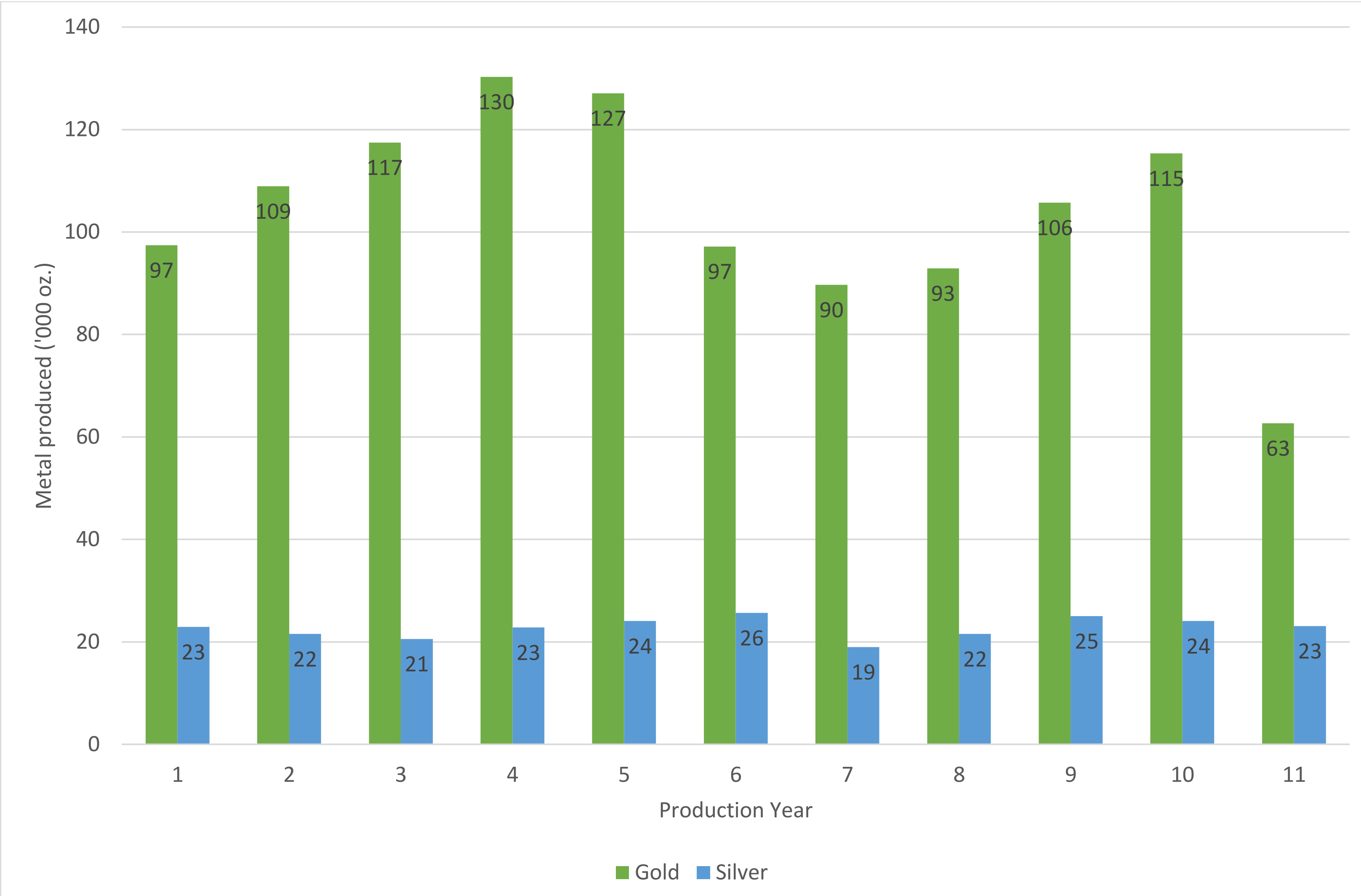
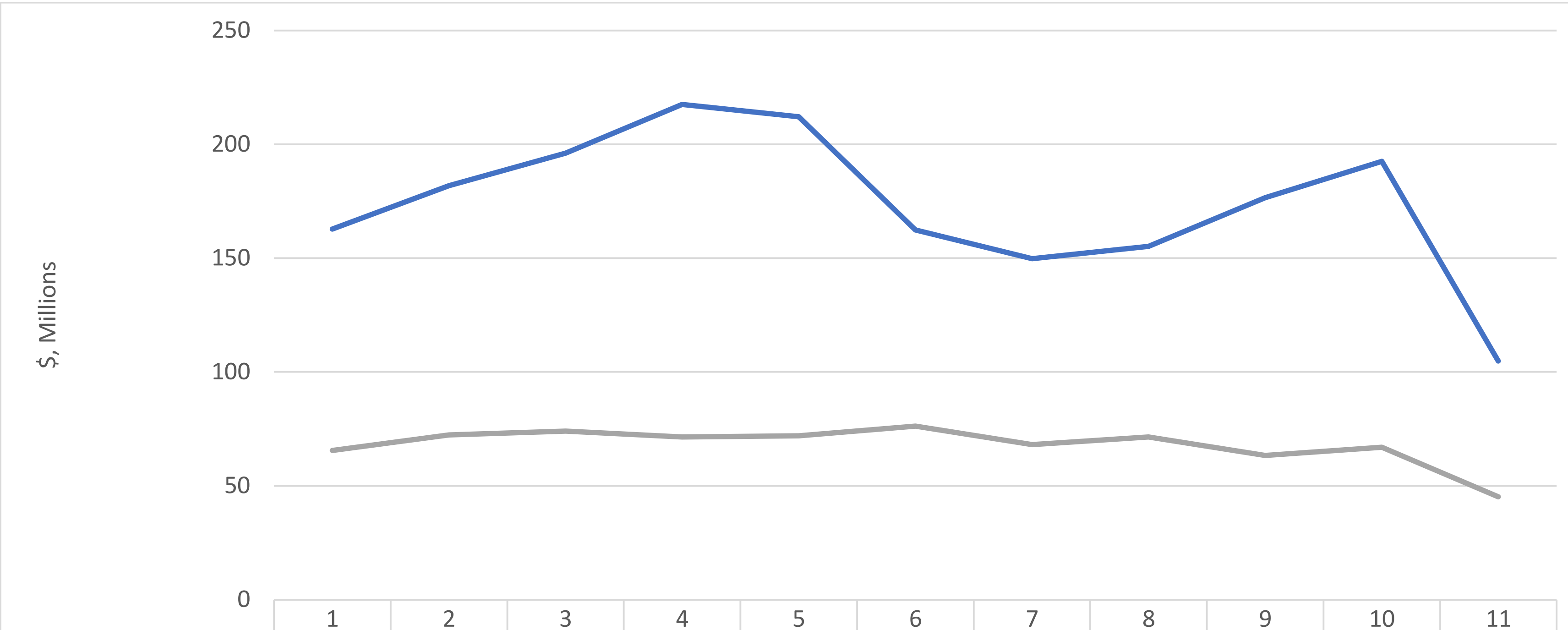


Figure 22-1 LOM Gold and Silver Production

The following graph, 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 and G&A costs

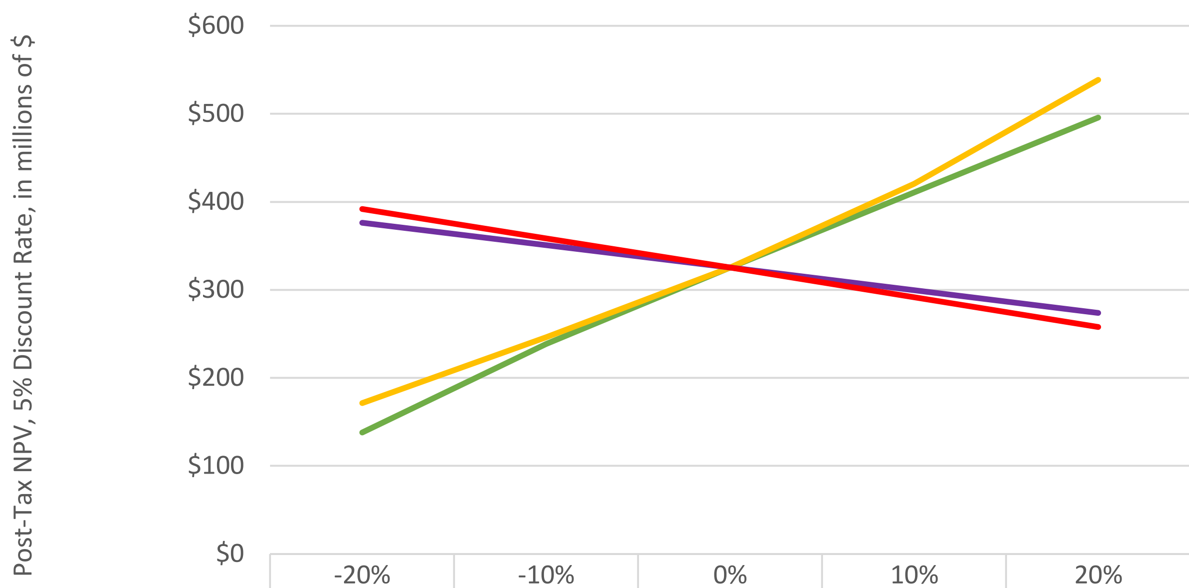


	1	2	3	4	5	6	7	8	9	10	11
Net receipts	163	182	196	217	212	162	150	155	177	193	105
Operating Costs	66	72	74	71	72	76	68	71	63	67	45

Figure 22-2 Net Receipts vs. Operating Costs

The following graphs show, for each case, the economic result sensitivities to:

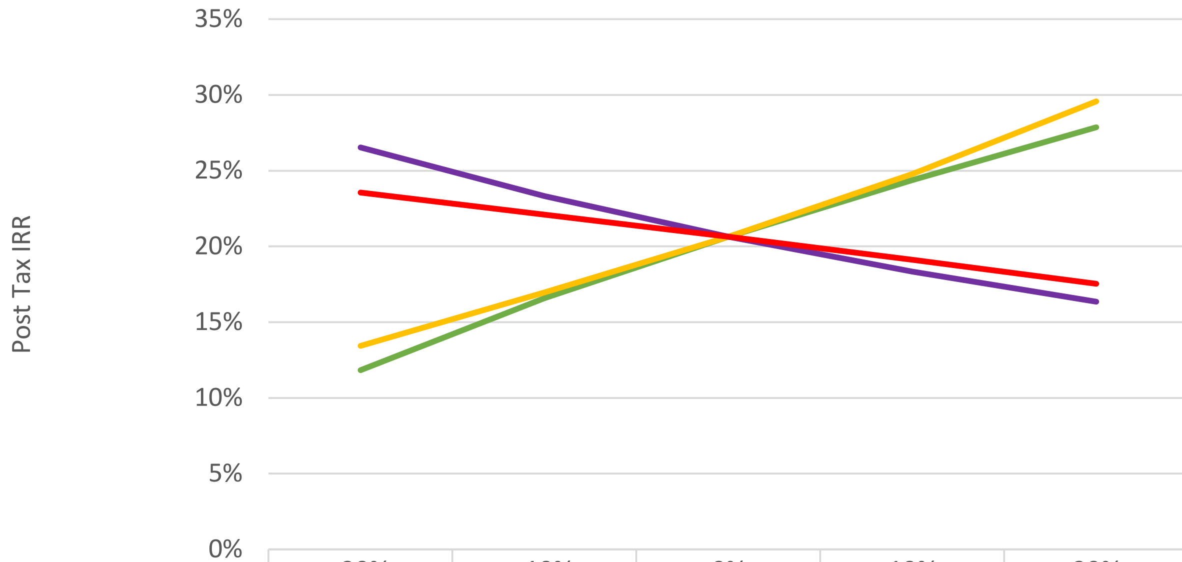
- Gold Price
- Foreign Exchange Rate
- Project Capital Costs and
- Operating Costs (mining, processing, TMF and G&A costs)



	-20%	-10%	0%	10%	20%
Gold Price	138	239	325	411	496
Exchange Rate	171	246	325	421	539
Capital Costs	376	351	325	300	274
Operating Costs	392	359	325	292	258

Parameter Variance

Figure 22-3 Sensitivity of Post-Tax NPV (5% Discount Rate) to various project inputs



	-20%	-10%	0%	10%	20%
Gold Price	12%	17%	21%	24%	28%
Exchange Rate	13%	17%	21%	25%	30%
Capital Costs	27%	23%	21%	18%	16%
Operating Costs	24%	22%	21%	19%	18%

Parameter Variance

Figure 22-4 Sensitivity of Post-Tax IRR to various 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 11-year mine life are shown in
Table **22-3**.

Table 22-3 Components of the Various Taxes

Tax Component	LOM Amount (M$)
Corporate Tax (Federal)	53.7
Corporate Tax (Provincial)	43.0
Less Investment Tax Credit (ITC)	(2.1)
Provincial Resource Tax	50.7
Total Taxes	**145.3**

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 BC provincial income taxes are calculated using the currently enacted corporate rates of 15% for federal and 12% for BC.

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.

Beginning 2020, 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. As SMG does not anticipate commercial production prior to 2020, the tax model adopts the provision for all its mining assets.

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.

23.0 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 194 million 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).

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.

24.0 Other Relevant Data and Information

No additional relevant information or data to disclose.

25.0 Interpretations and Conclusions

A PEA open pit mine plan has been developed using Mineral Resource estimates. It is the opinion of the QP's that the PEA shows positive economic viability.

25.1 Mineral Tenure, Surface Rights, Water Rights, Royalties and Agreements

- The mineral tenure held is valid and sufficient to support the Mineral Resources.
- Surface rights will be required before operations.
- Royalties are payable to third parties.
- There are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the Property that have not been discussed in this Report.

25.2 Geology, Mineralization, Exploration

- The deposit is considered to be an example of a Sediment Hosted Vein gold mineralization.
- Knowledge of the deposit settings, lithologies, mineralization style and setting, and structural and alteration controls on mineralization is sufficient to support the Mineral Resource estimation.
- The quantity and quality of the lithological, geotechnical, collar and downhole survey data collected in the drill programs are sufficient to support Mineral Resource estimation.
- SMG has been drilling on the Property since 2005. To date SMG drilling totals about 190,000 m in 910 drillholes.
- In the general area of the resource the following drillholes have been completed:
  - 679 core drillholes from 2005 to 2012 inclusive
  - 174 RC drillholes from 2004 to 2006 and 2013 to 2018 inclusive.
- The Mineral Resource has been estimated using data from these drillholes.
- The sample security, sample preparation and analytical procedures during the exploration programs by SMG followed accepted industry practice appropriate for the stage of mineral exploration undertaken.
- Data verification has been extensively conducted by Spanish Mountain Gold, and no material issues have been identified by those programs.
- Data collected have been sufficiently verified that they can support Mineral Resource estimation and used for mine planning purposes.

25.3 Resource Conclusions

The mineral resource is suitable for a PEA study.

- The base case mineral resource contains 30 Mt of 0.60 g/t Au and 0.83 g/t Ag in the Measured category; 244 Mt of 0.46 g/t Au and 0.69 g/t Ag in the Indicated category; and 52 Mt of 0.37 g/t Au and 0.67 g/t Ag in the Inferred category, based on a 0.15 g/t Au cut-off.
- There is a total of 4.1 Moz. of Au in the Measured and Indicated categories, with an additional 619 koz. in the Inferred category.

25.4 Mining Conclusions

The evaluation of mining options available from this deposit indicates that:

- There are adequate Measured and Indicated class resources in the deposit to develop an open pit mine and supply a mill with 3.7 Mt of resource per year over an 11-year period.
- The mine plan supports the cash flow model and financials developed for the PEA.
- The open-pit contents are based on LG analyses where the ultimate pit limits are selected. The selected ultimate pit limits are derived from a gold price input below the base case of US$1,250/oz. and provide some margin to future changes to prices and costs in future studies and over the estimates in the Life of Mine Cashflow model.
- The open pits are split into five mineable phases, or pushbacks, to target higher grade resource earlier in the project.
- The mine plans are based on a subset of the Mineral Resources containing 39 Mt of mill feed at an average diluted gold grade of 1.00 g/t, an average diluted silver grade of 0.74 g/t, and a waste to resource strip ratio of 3.5.
- The pit layouts are typical of other open pit gold operations in Canada and the unit operations within the mining operating plan are proven to be effective for these other operations.
- The pit phasing and mine design provide a reasonable basis for the production schedule with adequate operating widths to meet the targeted mill feed rate of 3.7 Mt per year.
- Potential Waste Rock Storage Facilities have been identified near the deposit to contain waste rock from the pit.
- The unit operating cost information for the selected equipment fleet is based on operating statistics from similar fleets. The resultant mine operating costs are reasonable.

25.5 Process Conclusions

The evaluation of process options indicates that:

- Metallurgical testwork completed has been appropriate to the style of mineralization.
- Gold is predominantly associated with quartz and sulphide minerals (mainly pyrite).
- Ore is moderate to soft and requires grinding to a P_{80} of 180 µm for rougher flotation.
- Organic carbon is successfully removed from flotation concentrate using CMC as a suppressant during cleaning and re-cleaning flotation
- Scavenger gravity concentration of cleaner and recleaner tails recovers 50% of the gold in cleaner/recleaner tails while rejecting organic carbon to gravity tails.
- Gold mineralization is fine-grained particles requiring concentrate regrind to 35 µm prior to leaching.
- Overall gold recovery of 91%, and silver recovery of 27%, is achievable.
- The process plant is based on a 10,000 t/d throughput and a flowsheet design including thres-stage crushing, ball mill grinding, multistage flotation, scavenger gravity of cleaner tails, concentrate regrind, and CIL to produce doré. CIL tailings are treated with the SO2/Air process to destroy cyanide.

25.6 Tailings Conclusions

- Two tailings streams are produced: rougher tailings and cleaner/CIL tailings, which will be transported from the plant site to the TSF in separate pipelines. Each tailings stream will be deposited independently; the rougher tailings will be discharged along the TSF embankments to create tailings beaches and the cleaner tailings will be discharged subaqueously in the supernatant pond and progressively encapsulated by the rougher tailings.
- The TSF capacity will be approximately 39 Mt of tailings, 25 Mt of PAG/ML waste rock, plus the supernatant pond volume and freeboard allowances.
- The TSF embankments will be constructed using suitable waste rock and overburden (low permeability glacial till) from the open pit.

25.7 Costs and Economic Analysis

- Initial capital cost for construction of the Project is estimated to be $364 million. Sustaining capital requirements over the 11-year mine life are estimated to be $58 million.
- Operating costs for the Project are estimated at a unit cost $19.10/t milled.
- The post-tax NPV5% is $325 million, and post-tax IRR is 21%. The projected payback is 3.5 years.
- The all-in sustaining cost is US$549/oz. Au. The LOM gold production is estimated at 1.1 million oz.
- 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.

26.0 Recommendations

The positive conclusions of this PEA lead the authors to recommend that the Project should proceed towards a higher level of engineering study.

The following work items and studies listed in Table 26-1 are directly recommended following this PEA, to lead to a decision point on whether to complete a PFS, or to proceed directly to a FS. The approach is directed towards the eventual completion of a FS on the Spanish Mountain Gold Project.

Table 26-1 Recommended future studies

Item	Description	Estimated Budget (M$)
1	Condemnation drilling	$1.00
2	Refine Geologic and Resource Model	$0.05
3	Mine Plan Refinement and Trade-Offs	$0.25
4	Pit and Waste Geotechnical Analysis	$1.70
5	Pit Area Seismic and Hydrogeology Studies	$0.50
6	Waste Rock Geochemical Characterization	$0.50
7	Metallurgical Studies and Process Refinement	$0.15
8	TSF Site Investigations	$1.40
9	AIA Investigations along Powerline Corridor	$0.20
10	BC Hydro Facility Study	$0.70
11	LiDAR Study along Powerline Corridor	$0.25
12	Permitting Activities	$2.00

The estimated dollar amount for these items is not included in the Project capital estimate or economic analysis conducted for this PEA.

The following recommendations are intended for consideration in the test work above and for the eventual PFS/FS work to follow. A more detailed scope of work will need to be developed when the studies and tests in Table 26-1 are completed.

26.1 Resource Recommendations

Discovery and MMTS recommend that future studies consider the following elements:

- Condemnation drilling of the footprints identified for the WRSF's and site infrastructure should be carried out.
- The geological model should be updated to include fault modelling, refinement of geologic boundaries based on updated drill results, and an overall update to the resource model incorporating these changes.

- Further work should be carried out to determine the reasons and processes behind the higher Au grades for the RC drilling in order to determine the most representative drilling type for this deposit and site (groundwater) conditions.
- Items 1 and 2 in Table 26-1.

26.2 Mining Recommendations

MMTS recommends that future studies consider the following mine engineering elements:

- Gather the required field data to proceed to the next level of study, including:
  - further definition of open pit and waste pile geotechnical characteristics,
  - further waste rock characterization, and
  - site analysis for alternative waste rock and stockpile locations.
- Condemnation drilling of the footprints identified for the WRSF's and site infrastructure should be carried out.
- Updating of all mine planning work done for this PEA 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.).
- Items 3 to 6 in Table 26-1.

26.3 Process Recommendations

MMTS recommends that future studies consider the following metallurgical and process engineering elements:

- Complete preliminary process engineering and plant design.
- Trade off studies to assess staged flotation reactors.
- Testwork to test the potential improvement by introducing gravity concentrators to the mill circuit (mill cyclone underflow).
- Item 7 in Table 26-1.

26.4 Infrastructure and Tailings Recommendations

MMTS and KP recommend that future studies consider the following infrastructure engineering:

- Optimize the site general arrangement.
- Initiate geotechnical site investigations to identify suitable borrow sources for construction materials.
- Consult with BCHydro to optimize off-site infrastructure for electrical power supply to site.
- Initiate engineering studies for the water balance, water quality and water management on site.
- AIA Investigations along selected powerline corridor.
- A surficial geology study and geotechnical site investigations in the TSF area to refine the assumptions made for the PEA cost estimate.
  - The surficial geology study would include a desktop study of the area, followed by ground truthing, laboratory test work, and analysis to confirm the finding of the desktop study.

  - A geotechnical site investigation program would include geotechnical drilling, test pitting, laboratory testing, and seismic surveys supporting future designs of the site buildings, TSF and the water management pond.
  - The site investigation program would include geochemical and hydrogeological drilling to support environmental baseline studies, test pitting, seismic surveys, laboratory test work and analysis.
- Investigation of the use of cyclone sands in construction of the TSF.
- Continue with provincial and federal permitting process.
- Items 8 to 12 in Table 26-1.

27.0 References

AGP Mining Consultants Inc. (2010): Preliminary Economic Assessment for the Spanish Mountain Project, NI43-101 Technical Report for Spanish Mountain Gold Ltd., dated Dec 20, 2010; filed on SEDAR, 366 pp

Beattie, M.J.V. (2013): Grade Determination for Spanish Mountain Gold Project, internal report for Spanish Mountain Gold Ltd; March 21, 2013

Bloodgood, M.A. (1988): Geology of the Quesnel Terrane in the Spanish Lake Area, Central British Columbia (93A/11); BC Ministry of Energy, Mines and Petroleum Resources, Geological Fieldwork 1987, Paper 88-1, p.139-145

British Columbia Hydro and Power Authority (BC Hydro, 2013): Spanish Mountain Mine (SMM) Load Interconnection Project System Impact Study, January 13, 2013

British Columbia Ministry of Energy, Mines and Petroleum Resources, Assessment Reports: 6460, 6935, 8636, 9762, 11822, 14682, 15880, 17636, 24729, 26210, 26473, 26477, 27415, 28113, 28457, 29105, 30144, 32368, 34080

Buryak, V.A. (1998): Sukhoi Log – The World's Largest Gold-Ore Deposit

Campbell, K.V. (2011): Review of Structural Geology of the North zone, Spanish Mountain Project, Cariboo Mining District, BC; internal report for BGC Engineering; October 28, 2011, 22 pp

Canadian Institute of Mining, Metallurgy and Petroleum (CIM) (2003): Estimation of Mineral Resources and Mineral Reserves, Best Practice Guidelines: Canadian Institute of Mining, Metallurgy and Petroleum, November 23, 2003

Canadian Institute of Mining, Metallurgy and Petroleum (CIM) (2014): CIM Standards for Mineral Resources and Mineral Reserves, Definitions and Guidelines: Canadian Institute of Mining, Metallurgy and Petroleum, May, 2014

Canadian Securities Administrators (CSA) (2011): National Instrument 43-101, Standards of Disclosure for Mineral Projects, Canadian Securities Administrators, May, 2016

Caterpillar (2016): Caterpillar Performance Handbook, Edition 46, January 2016

Fier, N.E., Wright, F., Simpson, R., and Martin, S. (2009): Prefeasibility Study on the Quesnel River (QR) Mine, Quesnel, BC, Canada, NI 43-101 Technical Report for Barkerville Gold Mines Ltd., dated Dec 14, 2009; filed on SEDAR, 102 pp

Gilmour, W.R. (2012): Summary of the QC/QA procedures on the Spanish Mountain Property, internal report for Spanish Mountain Gold Ltd, August 1, 2012

Gilmour, W.R. (2014): Summary of the QC/QA procedures on the Spanish Mountain Property, internal report for Spanish Mountain Gold Ltd, April 1, 2014

Giroux, G.H. and Koffyberg, A. (2011): Amended Technical Report on an Updated Mineral Resource Estimate on the Spanish Mountain Gold Deposit, for Spanish Mountain Gold Ltd., dated November 30, 2011; filed on SEDAR, June 2012, 141 pp

Giroux, G.H. and Koffyberg, A. (2012): Technical Report on an Updated Mineral Resource Estimate on the Spanish Mountain Gold Deposit, for Spanish Mountain Gold Ltd., dated August 31, 2012; filed on SEDAR, 124 pp

Giroux, G.H. and Koffyberg, A. (2014): Technical Report on an Updated Mineral Resource Estimate on the Spanish Mountain Gold Deposit, for Spanish Mountain Gold Ltd., dated April 25, 2014; filed on SEDAR, 162 pp

Johnston, R.J. (2005): Report on 2004 Exploration on the Spanish Mountain Property, internal report for Minocord Exploration Consultants, April 2005

Johnston, R.J. (2006): Assessment Report on Reverse Circulation Drill Programme, August-November 2006, Spanish Mountain Property, Cariboo Mining District, BC, for Skygold Ventures Ltd and Wildrose Resources Ltd; April 25, 2006; Assessment Report 29105

Johnston, R.J. (2006): Assessment Report on Reserve Circulation Drill Programme, September-November 2005, Spanish Mountain Property, Cariboo Mining District, BC, for Wildrose Resources Ltd; April 25, 2006; Assessment Report 28457

Klipfel, P. (2005): Carlin and Sediment Hosted Vein Deposits - An Intriguing Case of Common Characteristics; Symposium 2005, Geological Society of Nevada, Vol. 1, p. 79 – 91

Klipfel, P. (2007): Geologic Evaluation of the Spanish Mountain Project, British Columbia; internal report for Skygold Ventures Ltd., 25 pp

Knight Piésold Consulting (2012): Spanish Mountain Gold Project, Updated Waste and Water Management Concept Development Study, December 20, 2012

Koffyberg, A. (2011): Assessment Report on the 2010 Drilling Program on the Spanish Mountain Property, Cariboo Mining Division, BC, for Spanish Mountain Gold Ltd; Assessment Report 32368

Koffyberg, A. (2013): Assessment Report on the 2012 Drilling Program on the Spanish Mountain Property, Cariboo Mining Division, BC, for Spanish Mountain Gold Ltd; Assessment Report 34080

Koffyberg, A. (2019): Assessment Report on the 2018 Core Drilling Program and Archaeological Impact Assessment on the Spanish Mountain Property, Cariboo Mining Division, BC, for Spanish Mountain Gold Ltd; February 29, 2019; Assessment Report 38205

Koffyberg, A. and Gilmour W.R. (2019): Assessment Report on the 2018 RC Drilling Program on the Spanish Mountain Property, Cariboo Mining Division, BC, for Spanish Mountain Gold Ltd; June 15, 2019; Assessment Report 38319

Lustig, G.N. and Darney R. (2006): 2005 Drilling Summary with Recommendations for Exploration on the Spanish Mountain Property, Cariboo Mining District, BC, for Wildrose Resources Ltd, July 2006

Ministry of Energy, Mines and Petroleum Resources, Mining and Minerals Division (2008), Health, Safety and Reclamation Code for Mines in British Columbia

Moose Mountain Technical Services (2012): Technical Report and Preliminary Economic Assessment of the Spanish Mountain Gold Project, Likely, BC; effective date December 18, 2012

Moose Mountain Technical Services (2019): Preliminary Economic Assessment for the Spanish Mountain Gold Property, Likely, British Columbia; effective date for PEA study – April 10, 2017; amended date of report – February 11, 2019

Montgomery, A.T. (2009): Summary Report on 2009 Exploration Activities on the Spanish Mountain Gold Project, Cariboo Mining Division, BC; internal report for Skygold Ventures Ltd; December 21, 2009

Page, J.W. (2003): Compilation Report: A summary of the Exploration Programs and Results, Spanish Mountain Property, British Columbia, for Skygold Ventures Ltd; April 4, 2003; report filed on SEDAR, 80 pp

Panteleyev, A., Bailey, D.G., Bloodgood, M.A., and Hancock, K.D. (1996): Geology and Mineral Deposits of the Quesnel River – Horsefly map area, central Quesnel Trough, British Columbia NTS map sheet 93A/5,6,7,11,12,13; 93B/9, 16; 93G/1; 93H/4; BC Ministry of Energy, Mines and Petroleum Resources, Bulletin 97, 156 pp. Map at scale 1:100,000

Peatfield, G.R., Giroux, G.H., and Singh, R. (2009): Updated Resources Estimation Report on the Spanish Mountain Gold Deposit, Cariboo Mining Division, British Columbia, for Skygold Ventures Ltd; NI -43-101 report, dated May 1, 2009; filed on SEDAR, 76 pp

Price, G. (2008); Spanish Mountain Project, Cariboo District, BC, Structural Interpretations, for Skygold Ventures Ltd; January, 2008

Rees, C.J. (1981): Western Margin of the Omineca Belt Boundary at Quesnel Lake, British Columbia, in Current Research Part A, Geological Survey of Canada Paper 81-1A, p. 223-226

Rhys, D.A., Mortensen, J.K., Ross, K. (2009): Investigations of orogenic gold deposits in the Cariboo Gold District, east-central British Columbia (parts of NTS 093A, H): Progress Report; in Geoscience BC Summary of Activities 2008, Report 2009-1, p. 49-74

Ross, K.V. (2006): Petrographic Study of the Spanish Mountain Project, Cariboo Mining District, British Columbia; internal report for Skygold Ventures Ltd., 64 pp

Schiarizza, P. (2016): Towards a regional stratigraphic framework for the Nicola Group: Preliminary results from the Bridge Lake – Quesnel River area. In Geological Fieldwork 2015, BC Ministry of Energy and Mines, BC Geological Survey Paper 2016-1, pp 13-30

Singh, B. (2008): Technical Report on the Spanish Mountain Gold Property, for Skygold Ventures Ltd; NI 43-101 report, amended March 27, 2008; filed on SEDAR, 92 pp

Singh, B. And Stevens A. (2008): 2007 Assessment Report on the Spanish Mountain Property, Cariboo Mining District, for Skygold Ventures Ltd; December 1, 2008; Assessment Report 30144

Smee, B.W. (1998): Overview of Quality Control Procedures Required by Mineral Exploration Companies, in Workshop on Quality Control Methods in Mineral Exploration, Association of Exploration Geochemists, January 25-26, 1998, Vancouver, BC

SRK Consulting (2012): Technical Memo regarding ML/ARD Block Model Criteria, January 16, 2012.

Stantec (2012): Preliminary Economic Assessment Report, Spanish Mountain Mine Project, Interconnection Transmission Line to BC Hydro Power System, September 12, 2012.

Struik, L.C. (1983): Bedrock Geology of Spanish Lake (93A/11) and parts of Adjoining Map Areas, Central British Columbia. Geological Survey of Canada, Open File Map 920, 1: 50,000

Tetra Tech (2012): Technical Report and Preliminary Economic Assessment of the Spanish Mountain Gold Project, Likely BC; for Spanish Mountain Gold Ltd., dated Dec 18, 2012; filed on SEDAR, 272 pp

Tipper, H.W. (1971): Multiple Glaciation in Central British Columbia, in Canadian Journal of Earth Sciences, vol. 8, p. 743-752