NI 43-101

PRELIMINARY ECONOMIC ASSESSMENT FOR THE SPANISH MOUNTAIN GOLD PROPERTY

Likely, British Columbia, Canada

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

Effective Date for PEA Study: April 10, 2017
Effective Date for Resource Estimate: October 3, 2016

Submitted to:
Spanish Mountain Gold Ltd.

May 17, 2017

Moose Mountain Technical Services

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DATE & SIGNATURE PAGES

Herewith, our report entitled "Preliminary Economic Assessment for the Spanish Mountain Gold Property" dated May 17, 2017.

"*Marc Schulte*" {signed and sealed}

Signature of Member
Marc Schulte, P.Eng.

Signing Date: 17th of May, 2017
Effective Date: 10th of April, 2017

Moose Mountain Technical Services
Associate

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List of Abbreviations and Acronyms used throughout the report:

- *#/No./n Number (count)*
- *3D Three Dimensional*
- *3DBM 3D Block Model*
- *AAS Atomic Absorption Spectrophotometer*
- *Ag Silver*
- *Ai Abrasion Index*
- *AIA Archaeological Impact Assessment*
- *AISC All-in Sustaining Costs*
- *ALS ALS Global Minerals Lab*
- *ANFO Ammonium Nitrate and Fuel Oil*
- *AP Acid Potential*
- *ARD Acid Rock Drainage*
- *As Arsenic*
- *ASTM American Society for Testing and Materials*
- *Au Gold*
- *AUTOT Whole Block Diluted Gold Grades*
- *Az. Azimuth*
- *BC British Columbia*
- *BCWQG BC Water Quality Guidelines*
- *BGC BGC Engineering*
- *BMP Best Management Practices*
- *BWi Ball Mill Bond Work Index*
- *C Carbon*
- *Ca Calcium*
- *CAPEX Capital Expenditure*
- *CCA Capital Cost Allowance*
- *CCME Canadian Council of Ministers of the Environment*
- *CDE Canadian Development Expense*
- *CDN CDN Resources Labs Ltd.*
- *CEA Cumulative Expenditure Account*
- *CEAA Canadian Environmental Assessment Agency*
- *CEE Canadian Exploration Expense*
- *CIL Carbon in Leach*
- *CIM Canadian Institute of Mining*
- *CMC Carboxymethyl cellulose*
- *CSA Canadian Securities Administration*
- *CTCA Cumulative Tax Credit Account*
- *Cu Copper*
- *CWi Crushing Work Index*
- *DDH Diamond Drillhole*
- *DFO Fisheries and Oceans Canada*
- *Dist. Distance*
- *EA Environmental Assessment*
- *EAO BC Environmental Office*
- *ECC Environment and Climate Change Canada*
- *EGRG Extended Gravity Recoverable Gold*
- *Elev. Elevation*
- *EM Electromagnetic*
- *EPCM Engineering, Procurement, and Construction Management*
- *Fe Iron*
- *FS Feasibility Study*
- *FSR Forrest Service Road*
- *G&A General and Administration*
- *G&T G&T Metallurgical Services*
- *GCL Giroux Consultants Ltd.*
- *GME General Mine Expense*
- *GPS Global Positioning System*
- *GSC Geological Survey of Canada*
- *HDPE High Density Polyethylene*
- *HQ/HQ3 Drillhole Size, 96mm diameter*
- *HVAC Heating, Ventilation, and Air Conditioning*
- *ICP Inductively Coupled Plasma Mass*
- *ID Identification*
- *IEC International Electrotechnical Commission*
- *IP Induced Polarization*
- *IRR Internal Rate of Return*
- *ISO International Standards Organization*
- *IT Information Technology*
- *ITC Investment Tax Credit*
- *KP Knight Piésold*
- *LG Lerchs-Grossman*
- *LOM Life of Mine*

- *MIBC* *Methyl Isobutyl Carbinol*
- *ML* *Metal Leaching*
- *MMER* *Metal Mining Effluent Regulations*
- *MMTS* *Moose Mountain Technical Services*
- *Mo* *Molybdenum*
- *MS* *Mass Spectrometry*
- *MTO* *Mineral Titles Online*
- *NaCN* *Sodium Cyanide*
- *NAD* *North America Datum*
- *NI 43-101* *National Instrument 43-101*
- *NPAG* *Non-Potentially Acid Generating*
- *NP* *Neutralization Potential*
- *NPV* *Net Present Value*
- *NQ/NQ2* *Drillhole Size, 75.7mm diameter*
- *NRCan* *Natural Resources Canada*
- *NSP* *Net Smelter Price*
- *NSR* *Net Smelter Return*
- *NTS* *National Topographic Survey*
- P_{80} 80^{th} *percentile*
- *PAG* *Potentially Acid Generating*
- *PAX* *Potassium Amyl Xanthate*
- *PEA* *Preliminary Economic Assessment*
- *PFS* *Preliminary Feasibility study*
- *pH* *Potential Hydrogen*
- *QA* *Quality Assurance*
- *QC* *Quality Control*
- *QP* *Qualified Person*
- *RC* *Reverse Circulation*
- *ROG* *Ropes of Gold*
- *ROM* *Run of Mine*
- *RQD* *Rock Quality Designation*
- *RWi* *Rod Mill Work Index*
- *S* *Sulphur*
- *S/R* *Strip Ratio*
- *SAG* *Semi-Autonomous Grinding*
- *SCC* *Standards Council of Canada*
- *SG* *Specific Gravity*
- *SGS* *SGS Mineral Services*
- *SHV* *Sediment Hosted Vein*
- *SI* *International System of Units*
- *SIS* *System Impact Study*
- *SMC* *SAG Mill Comminution*
- *SMG* *Spanish Mountain Gold*
- SO_4 *Sulfate*
- *SP* *Stockpile*
- *SRK* *SRK Consulting*
- *TMF* *Tailings Management Facility*
- *TOC* *Total Organic Carbon*
- *TRIM* *Terrain Resource Information Management*
- *UHF* *Ultra-High Frequency*
- *UTM* *Universal Transverse Mercator*
- *WRSF* *Waste Rock Storage Facility*
- *Wt.* *Weight*
- *XRF* *X-Ray Fluorescence Spectrometer*

List of Units used throughout the report:

- *$* *Canadian fund dollars*
- *%* *percent*
- *º* *degrees*
- *µm* *micrometers*
- *a* *year(s)*
- *BCM* *bank cubic meters*
- *C$* *Canadian fund dollars*
- *cm* *centimeters*
- *d* *day(s)*
- *E* *easting*
- *g* *grams*
- *g/t* *grams per tonne*
- *h* *hour(s)*
- *ha* *hectares*
- *k* *thousands*
- *kg* *kliogram*
- *km* *kilometers*
- *koz* *thousands of ounces*
- *kt* *thousands of tonnes*
- *kV* *kilovolts*
- *kWh* *kilowatt hours*
- *L* *Liter*
- *M* *millions*
- *m* *meters*
- m^2 *square meters*
- m^3 *cubic meters*
- *mm* *millimeters*
- *Ma* *millions of years ago*
- *masl* *meters above sea level*
- *min* *minutes*
- *Mt* *millions of tonnes*
- *MW* *megawatt*
- *N* *northing*
- *Oz* *ounces*
- *ppb* *parts per billion*
- *ppm* *parts per million*
- *Q* *quarter of year*
- *t* *tonnes (metric)*
- *t/a* *tonnes per annum*
- *t/d* *tonnes per day*
- *US$* *US fund dollars*

1.0 Summary

1.1 Introduction

Spanish Mountain Gold Ltd. (SMG) retained Moose Mountain Technical Services (MMTS) to prepare a National Instrument 43-101 (NI 43-101) technical report and preliminary economic assessment (PEA) for the Spanish Mountain Gold Project (the Project). The Spanish Mountain Gold Property (the Property) involves the development of a gold deposit located in southcentral 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 Property is situated between Quesnel Lake and Spanish Lake; its centre is located at approximately latitude 52˚ 34' north and longitude 121˚ 28' west. The gold concentrator for the Project has been designed to process a nominal 7,300,000 t/a (or 20,000 t/d) of gold and silver bearing material from an open pit operation, and will produce gold-silver doré as a final product.

All currency amounts are referred to in Canadian dollars ($ or C$) unless otherwise indicated. All measurements are in metric units unless otherwise indicated.

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

Table 1-1 General PEA Results

Description	Unit	Amount
Pit Delineated Mineral Resources (Measured + Indicated)	Mt	178
Pit Delineated Mineral Resources (Inferred)	Mt	21
Life-of-mine (LOM)	years	24
Milling Rate	t/d	20,000
Strip Ratio	t/t	1.4
Total Project Initial Capital Cost	$ million	507
Average Overall Operating Cost	$/t milled	9.92
Gold Price	US$/oz	1,250
Pre-Tax Net Present Value (NPV) at 5% Discount Rate	$ million	597
Pre-Tax Internal Rate of Return (IRR)	%	21
After-Tax Net Present Value (NPV) at 5% Discount Rate	$ million	482
After-Tax Internal Rate of Return (IRR)	%	19
Capital Payback Period	years	3.7

Note: Mineral resources which are not mineral reserves do not have demonstrated economic viability. Inferred mineral resources have a high degree of uncertainty as to their existence, and a great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred resource will ever be upgraded to a higher category.

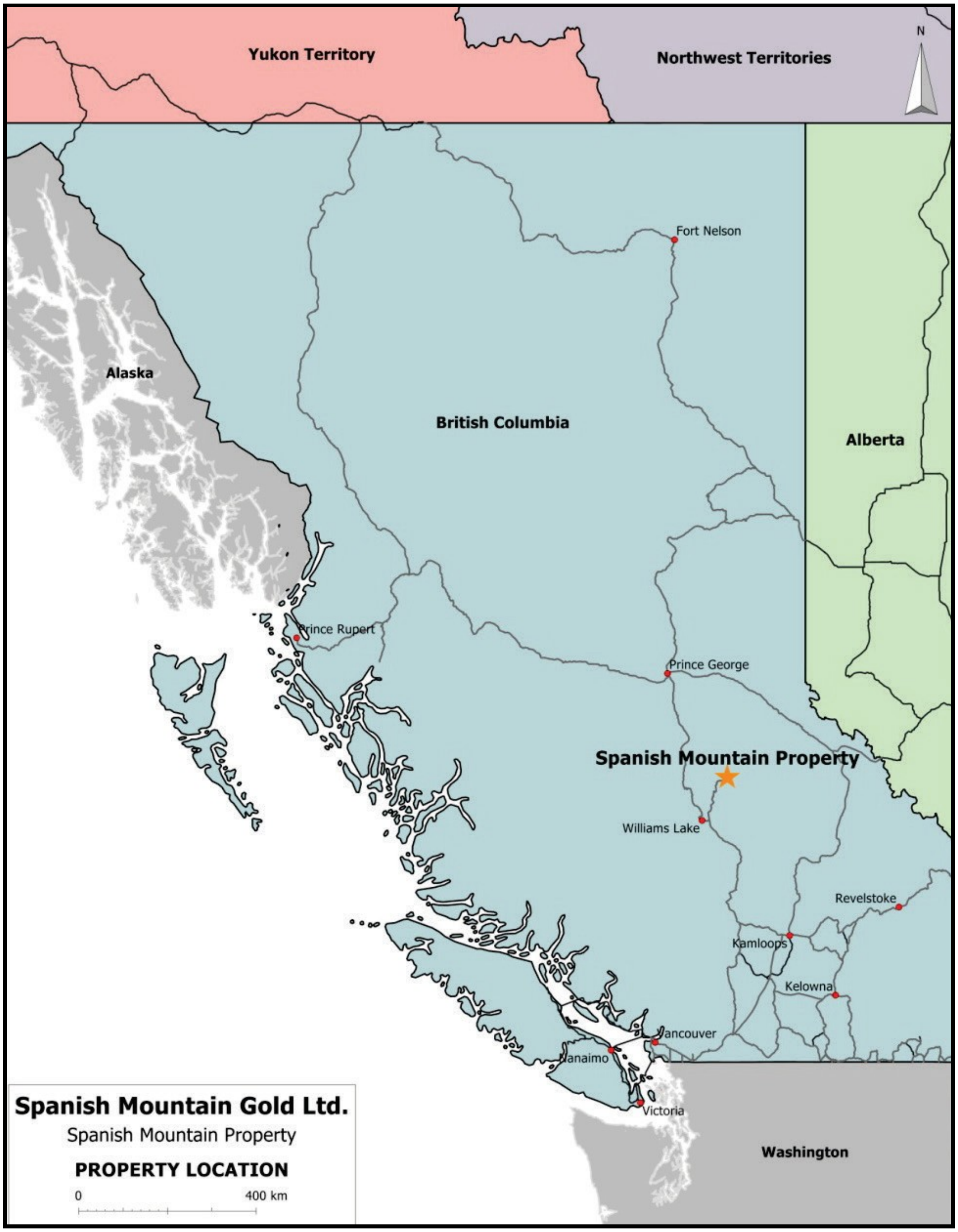


Figure 1-1 Property Location Map

MMTS worked with other consulting companies that took responsibility for various portions of the study. The areas of responsibility for each consultant are:

- Discovery Consultants (Discovery) – property description and location, accessibility, climate, and physiology, history, geological setting and mineralization, deposit types, exploration, drilling, sample preparation, data verification, and adjacent properties
- Giroux Consultants Ltd. (GCL) – mineral resource estimate
- Knight Piésold Ltd. (KP) – tailings, waste and water management, and environmental studies, permits and social or community impact

1.2 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 46 Mineral Titles Online (MTO) mineral claims, of which 20 are legacy claims. Of the 46 claims, 2 lie on the west side of Quesnel Lake; the other 44 form a contiguous block of claims covering an area of approximately 7,680 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).

1.3 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.

1.4 Mineral Resource Estimate

Mineral Resources for the Project were classified in accordance with CIM Definition Standards and NI 43-101 requirements. Table 1-4 summarizes the estimated Measured and Indicated Resource, and Table 1-5 summarizes the estimated Inferred Mineral Resource.

A gold cut-off grade of 0.15 g/t has been highlighted based on an economic assessment as a possible open pit cut-off.

Table 1-2 Spanish Mountain Gold 2017 Measured Resource

Au Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade > Cut-off		Contained Metal	
		Au (g/t)	Ag (g/t)	Oz. Gold	Oz. Silver
0.10	53,670,000	0.47	0.64	800,000	1,110,000
0.15	**45,730,000**	**0.53**	**0.66**	**770,000**	**970,000**
0.20	38,470,000	0.59	0.66	730,000	810,000
0.25	32,530,000	0.66	0.65	690,000	680,000
0.30	27,840,000	0.72	0.64	650,000	570,000
0.40	20,750,000	0.85	0.64	570,000	430,000
0.50	15,740,000	0.98	0.65	500,000	330,000
0.60	12,120,000	1.11	0.65	430,000	250,000
0.70	9,600,000	1.23	0.66	380,000	200,000
0.80	7,710,000	1.35	0.68	330,000	170,000
0.90	6,290,000	1.46	0.69	300,000	140,000
1.00	5,120,000	1.58	0.70	260,000	120,000

Table 1-3 Spanish Mountain Gold Indicated Resource

Au Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade > Cut-off		Contained Metal	
		Au (g/t)	Ag (g/t)	Oz. Gold	Oz. Silver
0.10	342,810,000	0.31	0.65	3,440,000	7,140,000
0.15	**260,800,000**	**0.37**	**0.67**	**3,110,000**	**5,650,000**
0.20	200,370,000	0.43	0.69	2,780,000	4,450,000
0.25	154,710,000	0.49	0.70	2,450,000	3,470,000
0.30	121,410,000	0.55	0.70	2,160,000	2,730,000
0.40	75,280,000	0.68	0.70	1,650,000	1,700,000
0.50	49,310,000	0.80	0.71	1,270,000	1,120,000
0.60	33,700,000	0.92	0.71	1,000,000	770,000
0.70	23,680,000	1.04	0.72	790,000	550,000
0.80	17,040,000	1.16	0.73	630,000	400,000
0.90	12,310,000	1.28	0.73	510,000	290,000
1.00	9,090,000	1.39	0.73	410,000	210,000

Table 1-4 Spanish Mountain Gold 2017 Measured plus Indicated Resource

Au Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade > Cut-off		Contained Metal	
		Au (g/t)	Ag (g/t)	Oz. Gold	Oz. Silver
0.10	396,480,000	0.33	0.61	4,240,000	7,790,000
0.15	**306,530,000**	**0.39**	**0.64**	**3,880,000**	**6,280,000**
0.20	238,840,000	0.46	0.66	3,510,000	5,030,000
0.25	187,240,000	0.52	0.67	3,140,000	4,020,000
0.30	149,260,000	0.59	0.68	2,810,000	3,240,000
0.40	96,030,000	0.72	0.69	2,210,000	2,130,000
0.50	65,040,000	0.85	0.69	1,770,000	1,450,000
0.60	45,810,000	0.97	0.69	1,430,000	1,010,000
0.70	33,280,000	1.10	0.69	1,170,000	740,000
0.80	24,750,000	1.22	0.70	970,000	560,000
0.90	18,600,000	1.34	0.71	800,000	420,000
1.00	14,210,000	1.46	0.71	670,000	320,000

Table 1-5 Spanish Mountain Gold 2017 Inferred Resource

Au Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade > Cut-off		Contained Metal	
		Au (g/t)	Ag (g/t)	Oz. Gold	Oz. Silver
0.10	691,530,000	0.23	0.59	5,070,000	13,140,000
0.15	**450,640,000**	**0.28**	**0.61**	**4,110,000**	**8,900,000**
0.20	307,410,000	0.34	0.63	3,320,000	6,250,000
0.25	203,740,000	0.39	0.65	2,580,000	4,240,000
0.30	136,250,000	0.45	0.66	1,980,000	2,900,000
0.40	61,590,000	0.59	0.69	1,160,000	1,360,000
0.50	32,180,000	0.72	0.69	740,000	710,000
0.60	18,410,000	0.85	0.67	500,000	400,000
0.70	11,280,000	0.99	0.68	360,000	250,000
0.80	7,590,000	1.10	0.68	270,000	170,000
0.90	4,920,000	1.24	0.68	200,000	110,000
1.00	3,400,000	1.37	0.67	150,000	70,000

Notes for Resource Tables:

- *Tonnages and Contained metals may not exactly equal individual tables due to rounding.*
- *This Mineral Resource Estimate was prepared by Gary Giroux, P.Eng. in accordance with CIM Definition Standards and NI 43-101, with an effective date of October 3, 2016.*
- *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Resource will ever be upgraded to a higher category.*
- *The estimate of Mineral Resources may be materially affected by environmental, permitting, legal or other relevant issues. The Mineral Resources have been classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves in effect as of the date of this Technical Report.*

1.5 Metallurgical Testing and Mineral Processing

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

Gold mineralization is fine-grained particles requiring concentrate regrind prior to leaching. Gold is predominantly associated with quartz and sulphide minerals (mainly pyrite). Preg-robbing organic carbon is successfully removed from flotation concentrate using CMC as a suppressant.

A gold flotation recovery of 95% has been achieved at an optimum grind size P_{80} of 184 µm. The test work supports a 90% overall gold recovery in the first years of production. Gold recovery drops to 87% in the subsequent years of production; in line with the reduced gold plant feed grade value. Overall LOM gold recovery is an estimated 89%. Overall silver recovery is assumed a constant 40%.

The 20,000 t/d process plant flowsheet design includes crushing, grinding, multistage flotation, scavenger gravity of cleaner tails, concentrate regrind, and CIL to produce doré. Comminution includes crushing, semi-autogenous grinding with pebble crushing, ball mill grinding, and cyclone classification. Cyclone overflow reports to a flotation circuit that includes a rougher stage with two open-circuit cleaner stages. A scavenging gravity concentration circuit for the cleaner and recleaner flotation tailings minimizes potential gold losses from the open flotation circuit configuration. Flotation tailings are pumped to a tailings management facility (TMF). Flotation concentrate is finely ground before leaching in a CIL circuit. Loaded carbon is transferred from the head CIL tank to an elution circuit where gold is stripped from the carbon and recovered from the elution solution by electrowinning. CIL tailings is pumped to cyanide detoxification, where cyanide levels are chemically reduced to acceptable environmental levels prior to disposal to the TMF, separate from the rougher tailings.

1.6 Mining and Pit Delineated Resources

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, using typical grade control methods and blast-hole sampling.

A PEA level mine operation design, approximately 14-year open pit production schedule, and cost model have been developed. The potential in-pit resource, based on a 0.15 g/t gold cut-off, is summarized in Table 1-6. The mine production schedule is described in Figure 1-2.

Table 1-6 Pit Delineated Resources

	Amount	Unit
Measured and Indicated	177,968	kt
Gold Grade	0.44	g/t
Measured and Indicated Insitu Gold	2,480	koz.
Silver Grade	0.67	g/t
Measured and Indicated Insitu Silver	3,837	koz.
Waste	257,102	kt
Strip Ratio	1.4	t/t
Inferred (included in Waste tonnage above)	21,226	kt
Gold Grade	0.30	g/t
Silver Grade	0.67	g/t

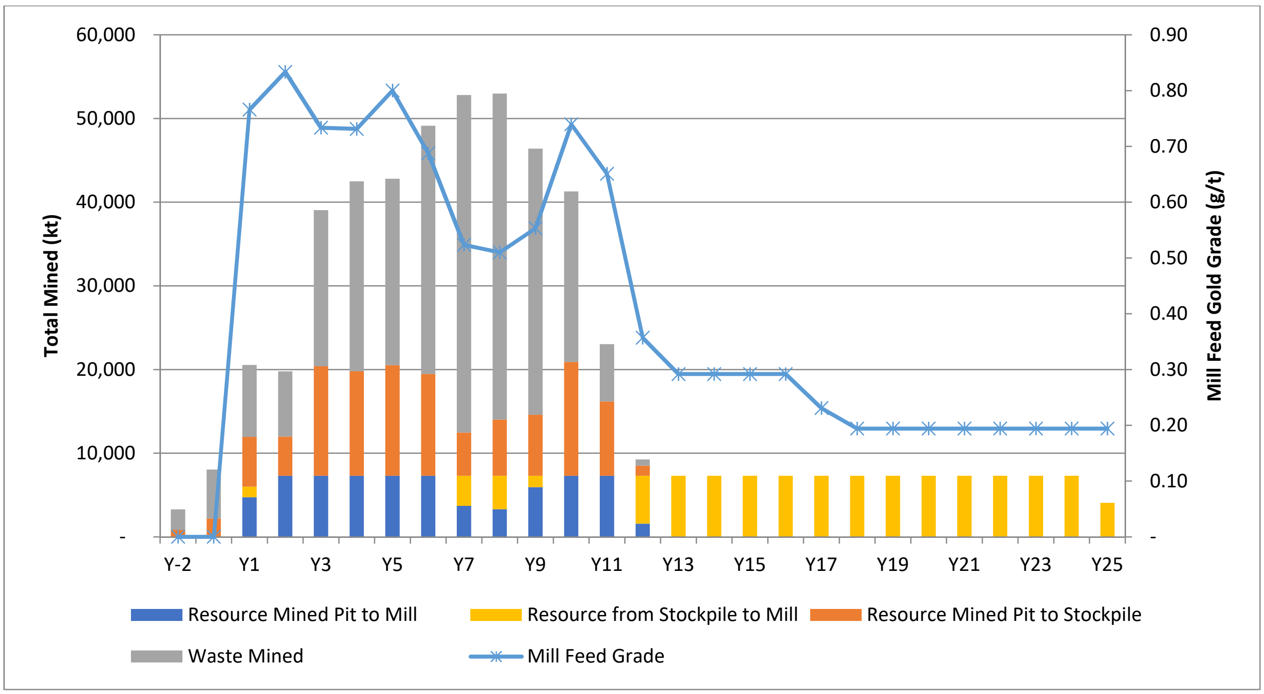


Figure 1-2 Mine Production Schedule

1.7 Project Infrastructure

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 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
- Tailing Management 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

A high-level layout of the on-site infrastructure is shown in Figure 1-3, which illustrates the overall project site layout.

The Project requires approximately 20 MW of peak load for 20,000 t/d operation demand. A new transmission line interconnecting the SMG site to BC Hydro’s power system is required to meet power requirement in operation. 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.

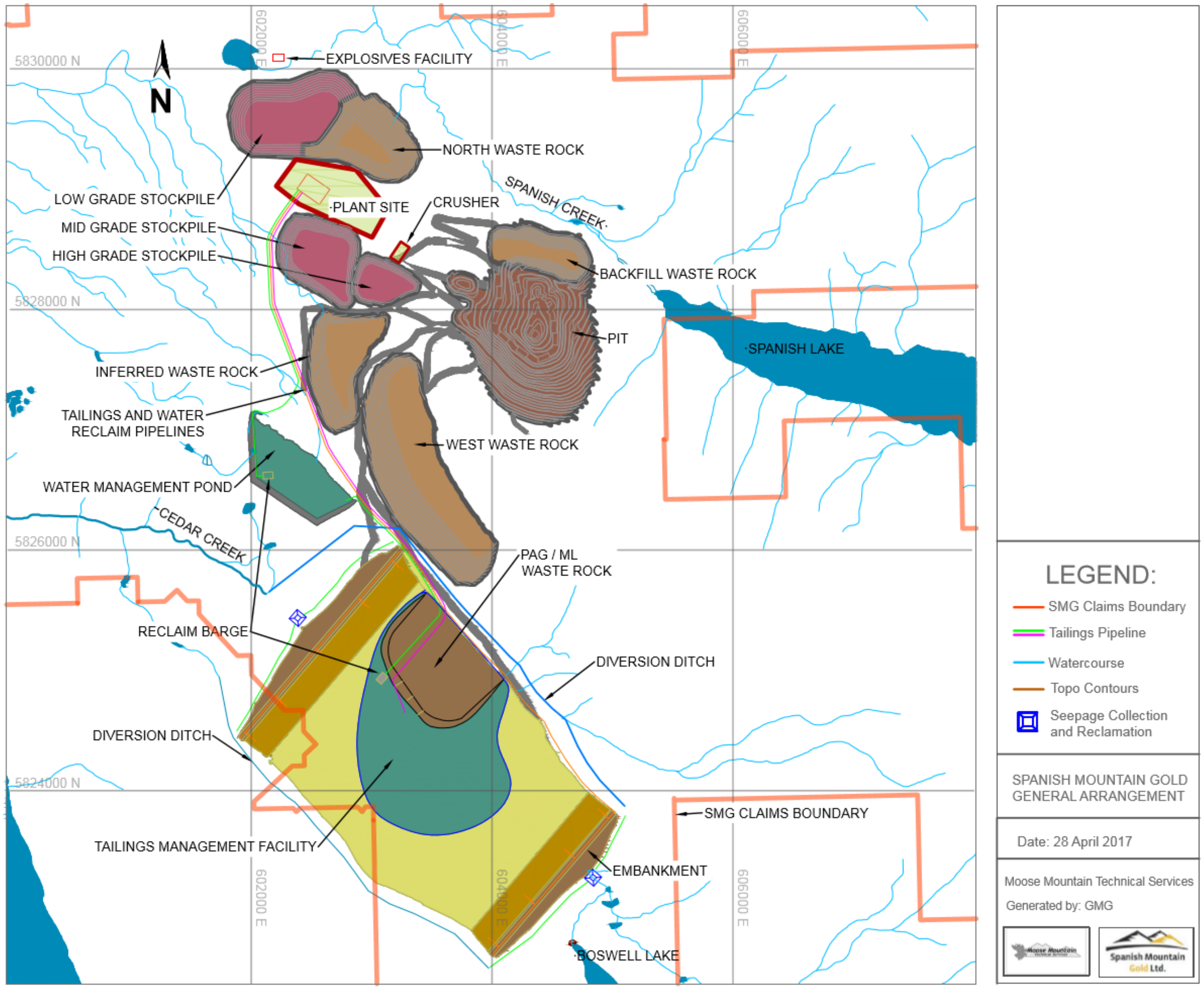


Figure 1-3 General Arrangement Layout

1.8 Waste and Water Management

The principal objective of the Tailings Management Facility (TMF) 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 TMF in separate pipelines. Each tailings stream will be deposited independently; the rougher tailings will be discharged along the TMF 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 TMF 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 TMF will be approximately 178 Mt of tailings, 32 Mt of PAG/ML waste rock, plus the supernatant pond volume and freeboard allowances.

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

1.9 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.

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 TMF 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 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.

1.10 Capital and Operating Costs

1.10.1 Capital Cost Estimate

The total estimated pre-production capital cost for the design, construction, and installation and commissioning for all facilities and equipment is shown in Table 1-7.

The accuracy of the estimate is ±40%. This study has been prepared with a base date of Q1 2017 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 1-7 Capital Cost Summary

Direct Costs	**Initial Capital Cost (M$)**
Overall Site	16.6
Open Pit Mining	97.3
Processing Plant (including Ore Handling)	140.0
Tailing Management Facility & Water Management	56.8
Environmental	12.0
On-Site Infrastructure	28.6
Off-Site Infrastructure	14.3
Direct Costs Sub-Total	**365.6**
Indirect Costs	
Project Indirects	84.6
Owner's Costs	5.8
Contingencies	51.1
Indirect Costs Sub-Total	**141.5**
Total Initial Capital Cost	**507.1**
Total Sustaining Capital Cost	**193.5**

1.10.2 Operating Cost Estimate

The unit costs summarized in Table 1-8 are based on an annual production rate of 20,000 t/d, and 365 d/a of operation.

Table 1-8 Operating Cost Summary

Area	Unit Cost
Mining ($/t mined)	$1.96
Mining ($/t milled)	$4.79
Processing ($/t milled)	$4.01
Tailings ($/t milled)	$0.05
G&A ($/t milled)	$1.09
Total ($/t milled)	**$9.94**

1.11 Economic Analysis

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

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

For the 25-year mine life and 178 Mt resource inventory, the following pre-tax financial parameters were calculated:

- 21% IRR
- 3.7-year payback on $507 million capital
- $597 million NPV at 5% discount value.

The following post-tax financial parameters were calculated:

- 19% IRR
- 3.7-year payback on $507 million capital
- $482 million NPV at 5% discount rate.

The following parameters are used for the financial analysis:

- Gold price of US$1,250/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:

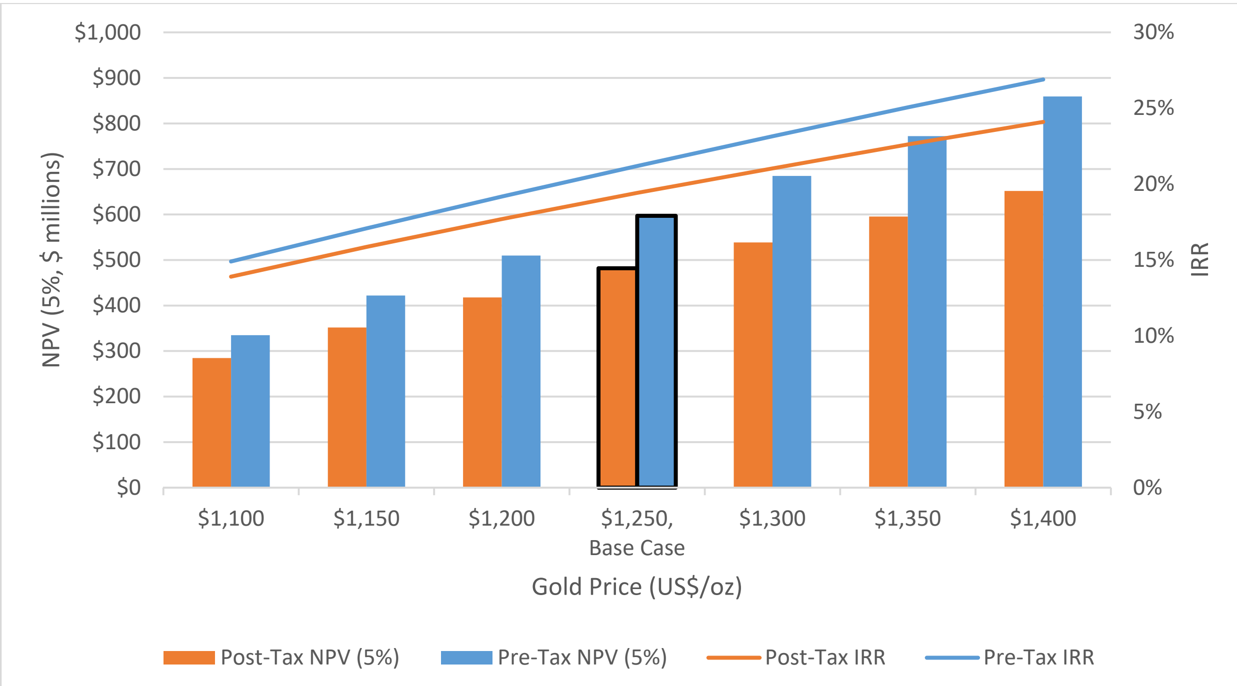


Figure 1-4 Project Economic Sensitivity

1.12 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 14-year open pit operation and a 24-year plant operation for life of mine production of 2.2M ounces of gold at an average grade of 0.44 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, GCL, 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. The authors, in writing this report use sources of information as listed in the references listed in Section 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.

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 chart:

Table 2-1 QP/Author Responsibility Chart

Qualified Person	Company	Sections of Responsibility
Bill Gilmour, P.Geo.	Discovery	4-12, 23
Gary Giroux, P.Eng.	GCL	14, 25, 26
Marc Schulte P.Eng.	MMTS	1-3, 15, 16, 18, 19, 21, 22 24-27
Tracey Meintjes, P.Eng.	MMTS	13, 17, 25, 26
Les Galbraith, P.Eng.	KP	18.4, 20

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

- Bill Gilmour P.Geo. conducted a site visit on August 23, 2013.
- Gary Giroux., P.Eng. conducted a site visit on June 29, 2011.
- Marc Schulte, P.Eng. conducted a site visit on October 6, 2011.
- Tracey Meintjes, P.Eng. has not conduced a site visit.
- Les Galbraith, P.Eng. conducted a site visit on October 4, 2012.

3.0 Reliance on Other Experts

Bill Gilmour has relied upon SMG management for information regarding the various option agreements and net smelter royalties. It was not within the scope of the Report to verify the legal status or ownership of the Property. Research into the mineral claim status was limited to the information available on British Columbia MTO website.

Marc Schulte has relied on Larry Yau of SMG for the financial model described in Section 22.0, Economic Analysis, communicated via spreadsheet on April 02, 2017.

4.0 Property Description and Location

4.1 Location

The Property is located in the Cariboo region of central British Columbia, approximately 6 km southeast of Likely and 66 km northeast of Williams Lake (Figure 4-1). The Property is situated between Quesnel Lake and Spanish Lake with the centre approximately at latitude 52° 35' north and longitude 121° 28' west. 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 604425 east and 5827900 north (Datum NAD83, Zone 10). It is located mainly within the mineral title 204667 as well as mineral titles 204225 and 204226.

Figure 4-1 Property Location

4.2 Description

The Property consists of 45 MTO mineral titles, of which 20 are legacy claims. These mineral titles form a contiguous block covering an area of approximately 7,621 ha (Figure 4-2). 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 6 overlying placer titles (1,964 ha) in the area (Figure 4-3).

The Property overlies district lots of several private home owners along the eastern side of Quesnel Lake and one, small isolated parcel (DL12083) at the northwest end of Spanish Lake (Figure 4-4). In addition, a third party(s) owns six placer leases (DL 12740 to 12745, Figure 4-4). Cedar Point Provincial Park is a small 8-hectare Class C park, located where Cedar Creek enters Quesnel Lake (Figure 4-4). Part of the Park underlies claim 517485.

4.3 Ownership

SMG, with offices at 1120 – 1095 West Pender Street, Vancouver, BC, owns all 45 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.

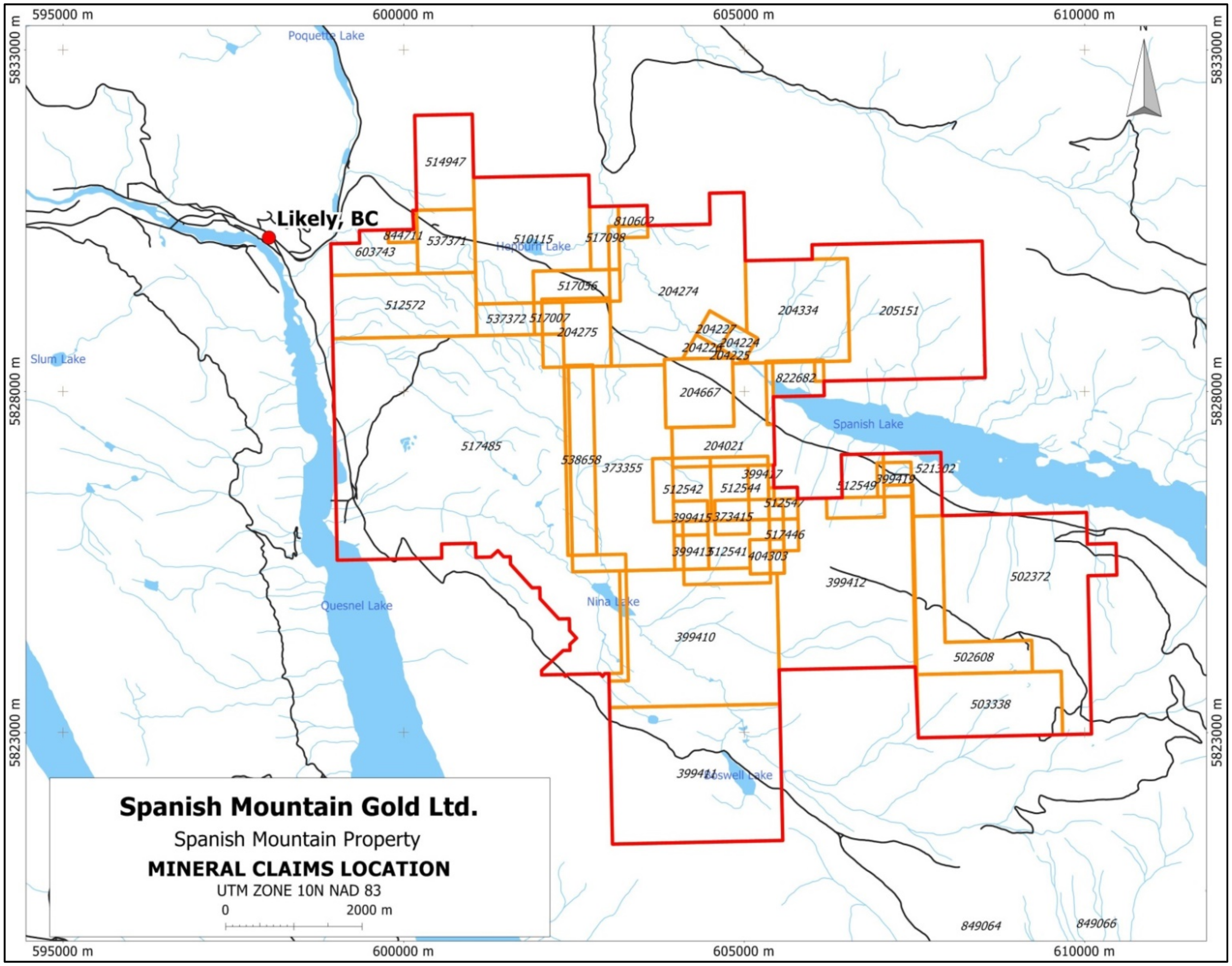


Figure 4-2 Mineral Claim Locations

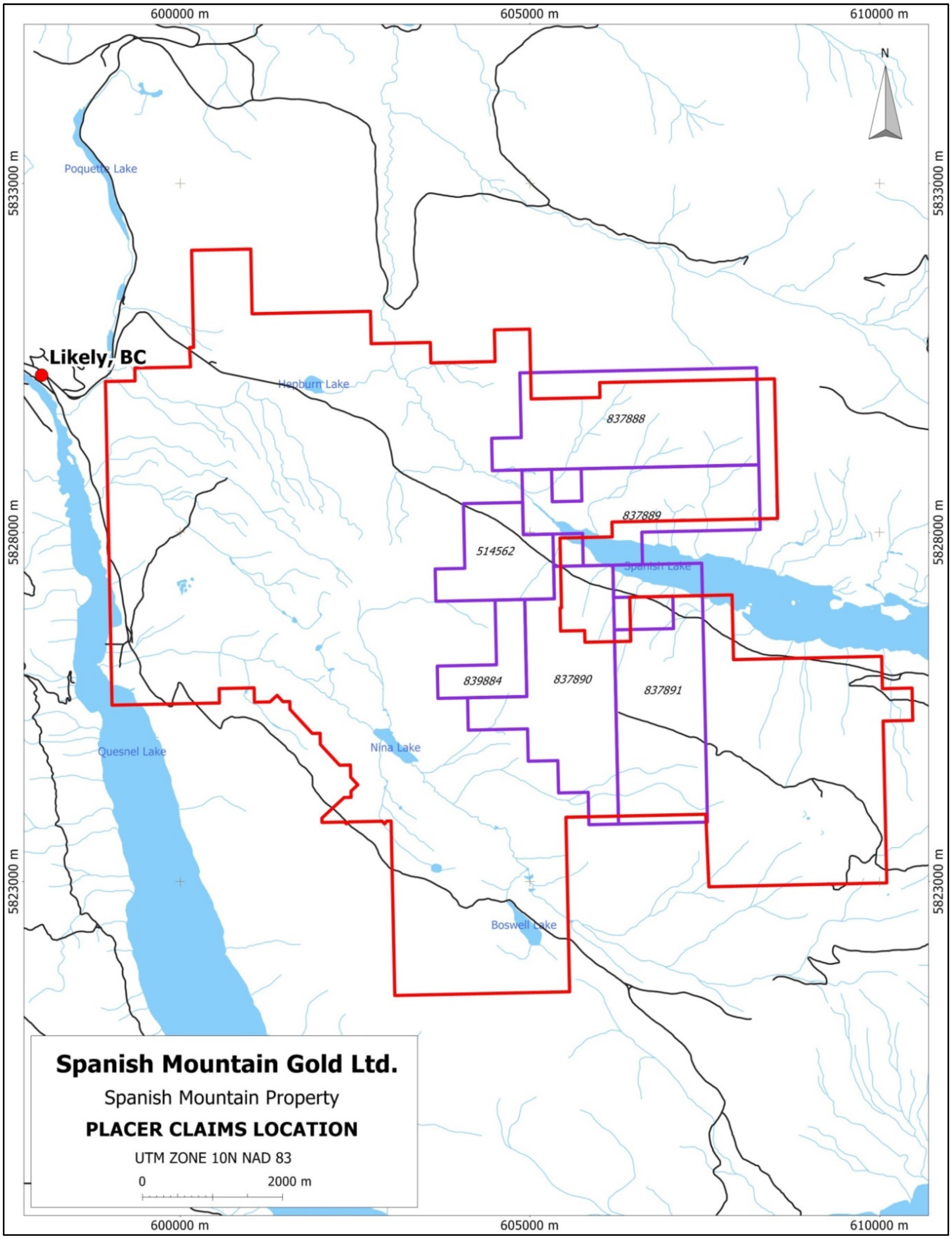


Figure 4-3 Placer Claim Locations

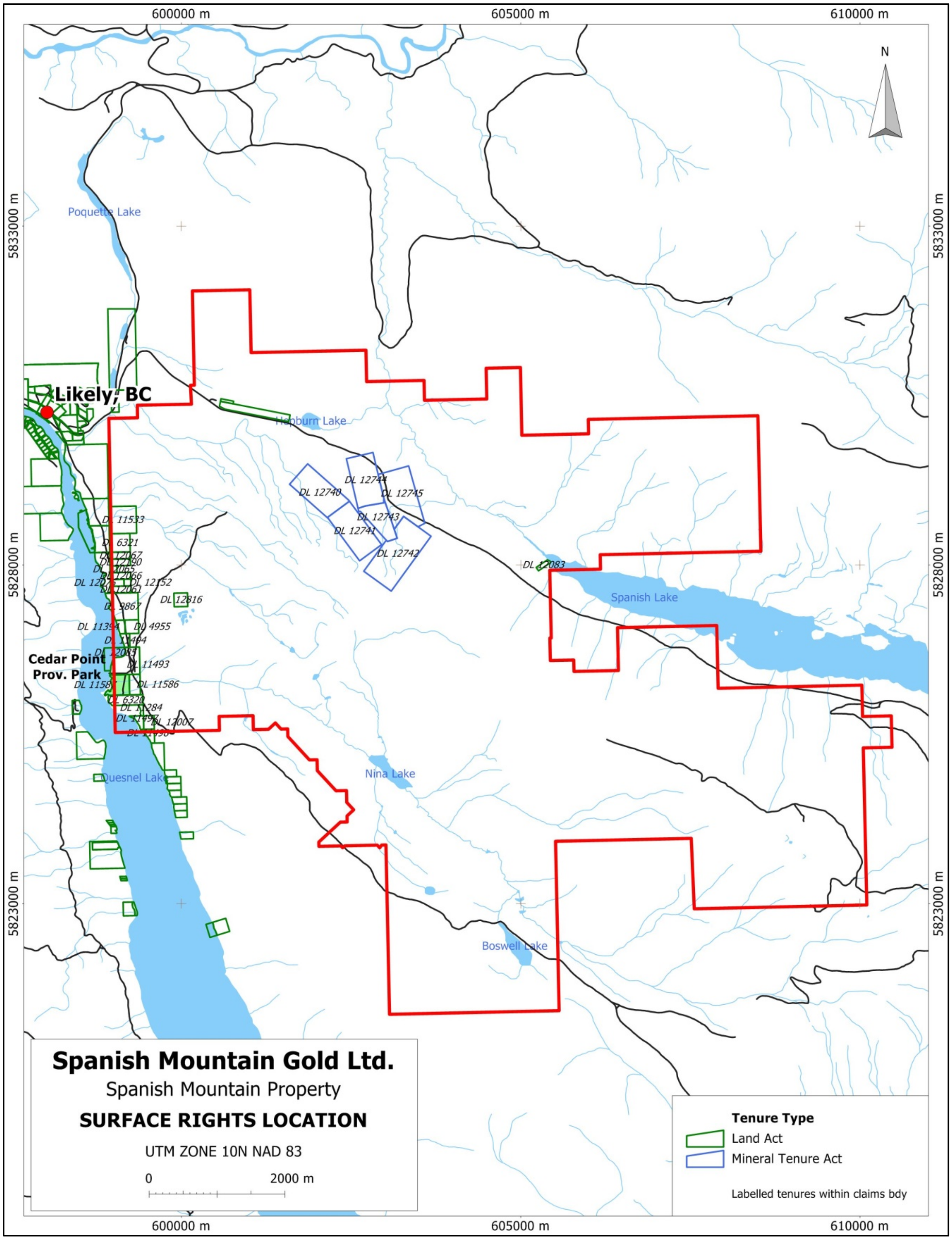


Figure 4-4 Surface Rights Locations

Table 4-1 Mineral Title Descriptions

Tenure Number	Area (ha)	Map Number	Registered Owner	Good To Date
204021	225.00	093A.053	Spanish Mountain Gold Ltd.	2023/Oct/31
204224	25.00	093A.053	"	2023/Oct/31
204225	25.00	093A.053	"	2023/Oct/31
204226	25.00	093A.053	"	2023/Oct/31
204227	25.00	093A.053/063	"	2023/Oct/31
204274	500.00	093A.053/063	"	2023/Oct/31
204275	100.00	093A.053/063	"	2023/Oct/31
204334	225.00	093A.053/063	"	2023/Oct/31
204667	100.00	093A.053	"	2023/Oct/31
205151	500.00	093A.053/063	"	2023/Oct/31
373355	450.00	093A.053	"	2023/Oct/31
373415	25.00	093A.053	"	2023/Oct/31
399410	500.00	093A.053	"	2023/Oct/31
399411	500.00	093A.053	"	2023/Oct/31
399412	500.00	093A.053	"	2023/Oct/31
399413	25.00	093A.053	"	2023/Oct/31
399415	25.00	093A.053	"	2023/Oct/31
399417	25.00	093A.053	"	2023/Oct/31
399419	25.00	093A.053	"	2023/Oct/31
403303	25.00	093A.053	"	2023/Oct/31
512541	117.89	093A.053	"	2023/Oct/31
502372	491.33	093A.053/054	"	2023/Oct/31
502608	157.23	093A.053/054	"	2023/Oct/31
503338	196.58	093A.053/054	"	2023/Oct/31
510115	274.82	093A.063	"	2023/Oct/31
512542	78.58	093A.053	"	2023/Oct/31
512544	78.58	093A.053	"	2023/Oct/31
512547	19.65	093A.053	"	2023/Oct/31
512549	78.58	093A.053	"	2023/Oct/31
512572	196.34	093A.063	"	2023/May/01
514947	117.76	093A.063	"	2023/Oct/31
517007	19.64	093A.063	"	2023/Oct/31
517056	58.90	093A.063	"	2023/Oct/31
517098	39.26	093A.063	"	2023/Oct/31
517446	19.65	093A.053	"	2023/Oct/31
517485	1335.78	093A.053	"	2023/May/01
521302	58.94	093A.053	"	2023/Oct/31
537371	78.52	093A.063	"	2023/Oct/31
537372	39.27	093A.063	"	2023/Oct/31
538658	117.86	093A.053	"	2023/May/01
603743	78.52	093A.063	"	2023/May/01
810602	19.63	093A.063	"	2023/May/01
822682*	78.56	093A.053	"	2023/Oct/31
844711	19.63	093A.063	"	2023/May/01

*Mineral titles in **red** are subject to the Mickle option agreement*
*Mineral title in **blue** is subject to the Wallster and McMillan option agreement*
*Mineral titles in **green** are subject to the Cedar Creek purchase agreement*
*Mineral titles in **purple** are subject to the Acrex purchase agreement*
* *Mineral title 822682 is converted from legacy claim 204727, which is subject to the Mickle option agreement*

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. The co-author Gilmour is not aware of any outstanding environmental issues that would be likely to delay or adversely affect the project.

5.0 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Access

The Property can be reached from the 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 three scheduled airlines that provide daily service with Vancouver, BC, and points north within BC.

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 15km 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.

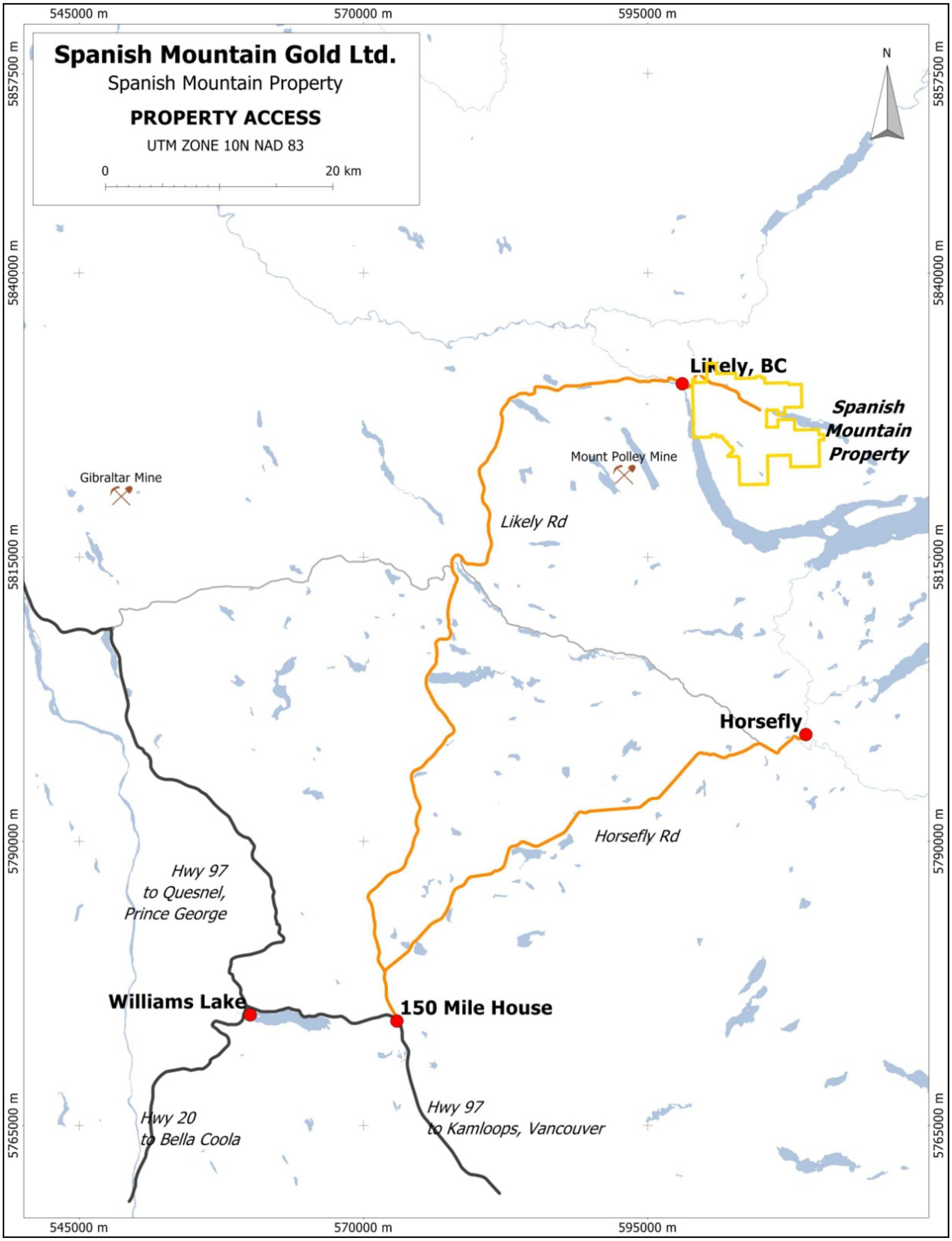


Figure 5-1 Property Access

5.5 Physiography

The Property covers an area of approximately 10 km north to south by 10 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 shore lines.

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.

6.0 History

The 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 Exploration

Year	Company	Work Done
2004	Wildrose Resources Ltd	2,506 m of RC drilling in 34 holes, 2,419 m of trenching, soil sampling *Discovery of disseminated mineralization in drilling
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

Year	Company	Work Done
		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
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 predominately sedimentary and volcanic rocks of middle to upper Triassic age, 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 Barkerville Subterrane of the Omineca Terrane. These include the Snowshoe Group (unit 7) and the Quesnel Lake Gneiss. Splays of the Eureka Thrust, including the Spanish Thrust, bisect the Spanish Mountain area.

East of the Spanish Thrust is the Crooked Amphibolite unit of the Slide Mountain Terrane, of Pennsylvanian to Permian age (unit 6), which overlies the Snowshoe Group in thrust fault contact. It consists of talc chlorite schists, amphibolites, serpentinites and ultramafic rocks. Slocan Group sediments unconformably overlie the Slide Mountain rocks.

The stratigraphy of the Quesnel Terrane in the Spanish Mountain area has been examined by Rees (1981), Struik (1983), and Bloodgood (1988). Panteleyev et al. (1996) have produced a geological compilation of the Quesnel River - Horsefly area. Nomenclature has varied for the rocks within the central part of the Quesnel Terrane, such as Quesnel River Group, Horsefly Group, Takla Group and Nicola Group; however, Panteleyev et al. assigned the term Nicola Group rocks as the most accurate usage. The Quesnel Terrane consists of a sedimentary package of black graphitic argillites, phyllitic siltstones, sandstones, limestones and banded tuffs (units 5a and 5c), and is weakly metamorphosed. The age of the Nicola Group, based on conodont fossils found south of Quesnel Lake, is Middle to Late Triassic. A narrow sequence of volcanic and volcaniclastic rocks (unit 5b), occurs as a discrete subunit within the sedimentary sequences.

Recent work by Schiarizza (2016) reassigns the unit 5c rocks north of Spanish Lake to the middle to upper Triassic Slocan Group, with the unit 5c rocks immediately south remaining as Nicola. The stratigraphic/structural relationship between the Nicola and Slocan Lake sedimentary rocks is uncertain. West of Spanish Lake the contact trends northwesterly and east of the lake trends southeasterly. The rock types within these two units are very similar, except that volcaniclastic sediments are restricted to the Nicola rocks.

The overlying Nicola Group volcanic rocks (unit 4c) are in depositional contact with the metasediments. This unit is mainly of alkalic composition, and has been divided into an older package of dark grey to green flows, pillow basalts, breccias and tuff, and a younger sequence of dark green to maroon flows, tuff, volcaniclastic sandstone and breccias, with minor limestone (unit 4b).

Overlying the alkalic basalts is a younger package of volcanic rocks consisting predominantly of basaltic and feldspathic rocks with derived volcaniclastic sediments (unit 4a). Rock types include volcanic breccias, lahars, crystal lithic tuffs, sandstones and conglomerates.

The region has been strongly affected by fold and thrust deformations, as described by Bloodgood (1988) and Rhys et al. (2009), with at least two main phases of deformation, referred to as D1 and D2. Phase D1 deformation consists of isoclinal folding associated with the development of thrust faults, including the Eureka Thrust. This event is associated with peak metamorphism and is thought to have occurred sometime between 174 and 139 Ma; that is, mid-Jurassic to Early Cretaceous (Rhys et al., 2009). Phase D2 deformation includes the Eureka Peak syncline, which refolds earlier folds, forming open folds, and associated foliation and thrust faults. Structurally late, although possibly long lived are north-northeastly trending faults that have offset earlier thrusts and structures. These faults are associated with late gold-bearing quartz veins in the district.

Metamorphic mineral assemblages are of sub-greenschist facies. Figure 7-1 shows the regional geology, based on the work by Panteleyev et al. (1996), and as shown on the website of the MapPlace, of the British Columbia Ministry of Energy and Mines. The legend is given on Figure 7-2.

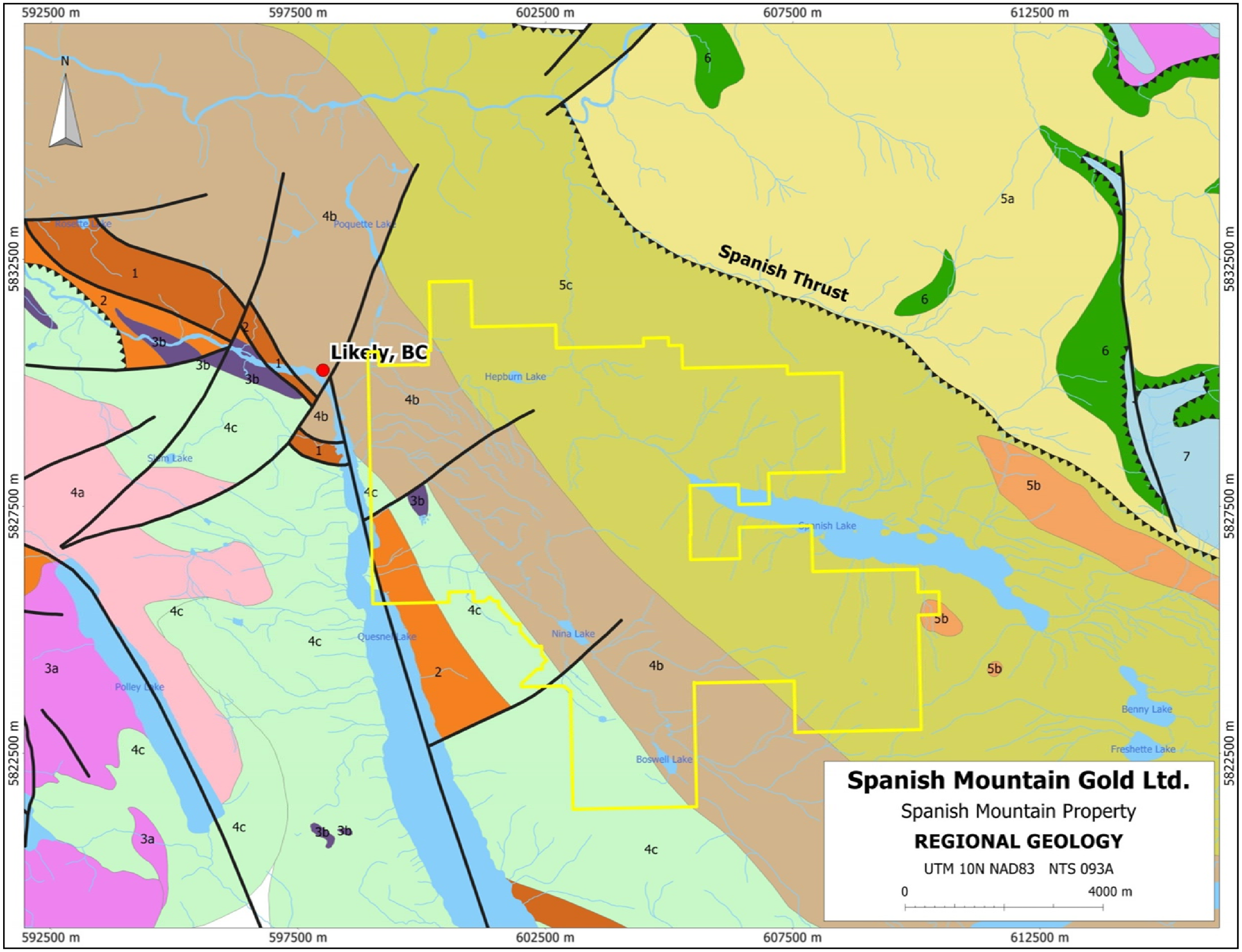


Figure 7-1 Regional Geology, Claims Boundary shown in yellow

LEGEND

SEDIMENTARY, METAMORPHIC & VOLCANIC ROCKS

CRETACEOUS

1 — Undivided sedimentary rocks, conglomerate, sandstone, shale.

JURASSIC

LOWER

2 — Argillite, greywacke, conglomerate turbidites; feldspathic sandstone and siltstone; minor limestone and calcareous siltstone.

TRIASSIC

UPPER (NICOLA GROUP)

4a — Polymict volcanic breccia containing clasts of latite, trachyte and intrusive equivalents; basalt flows and breccias; some felsic volcanic breccias and flows.

4b — Sandstone, siltstone, shale; slate and phyllite; bioclastic limestone; minor felsic tuff, tuffaceous argillite, basalt breccia and agglomerate.

4c — Pyroxene and pyroxene-hornblende basalt flows, breccias and tuffs; minor sandstone, siltstone, limestone and limestone breccia.

MIDDLE (NICOLA GROUP)

5a — Sandstone, siltstone, shale; slate and phyllite; bioclastic limestone; minor felsic tuff, tuffaceous argillite.

5b — Pyroxene and pyroxene-hornblende basalt flows, breccias and tuffs.

5c — Mixed volcaniclastic rocks, siltstone, sandstone and minor limestone.

CARBONIFEROUS-PERMIAN

CROOKED AMPHIBOLITE

6 — Ultramafic rocks- Serpentinite, sheared ultramafic rock, amphibolite, talc schist.

UPPER PROTEROZOIC-PALEOZOIC

SNOWSHOE GROUP

7 — Metasediments- quartzite, micaceous quartzite, schist, phyllite, gneiss, marble, amphibolite.

INTRUSIVE ROCKS

3a — Syenite, monzonite, monzodiorite, syenodiorite and diorite; minor nepheline syenite, clinopyroxenite, peridotite and gabbro.

3b — Syenite, monzonite, monzodiorite, syenodiorite and diorite; minor nepheline syenite, clinopyroxenite, peridotite and gabbro.

Map is based on Panteleyev et al. (1996)
and from the MapPlace .ca

Figure 7-2 Legend of the Regional Geology

7.2 Property Geology

Much of the information on the Property geology 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 on Figure 7-3, 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 (<1m), 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 <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.

The geology of the Property is given in Figure 7-3.

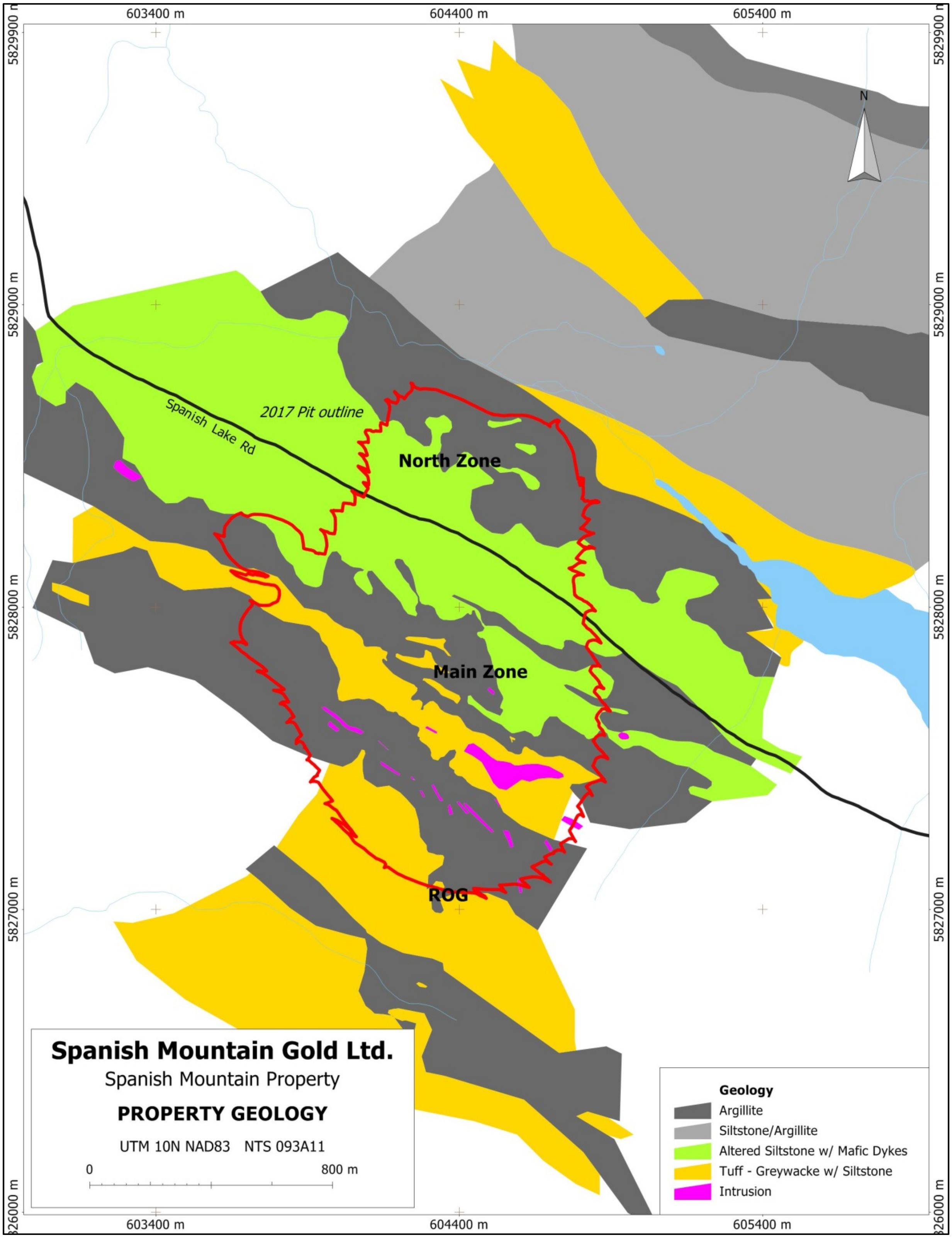


Figure 7-3 Property Geology, approximate pit outline shown in red

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 7km 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 Figure 7-1. 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 a 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 <5µm. 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.

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 deposits 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.

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 2004 to 2014. To date SMG drilling totals about 190,000 m in more than 740 drillholes. Thus, a summary is provided of the work done in these programs. Programs carried out before 2005 are summarized in Section 6.0 – Exploration History. Note that the 2005 to 2009 exploration programs carried out by SMG were done under its former name of Skygold Ventures Ltd.

Table 9-1 Summary of Exploration Programs

Year	Work Done
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

In 2005, SMG began core drilling and continued with RC drilling with joint venture partner Wildrose. A program totalling 7,746 m of core drilling (35 holes) and 3,377 m of RC drilling (34 holes) was carried out in the Main Zone and to a lesser extent in the South Zone, along with geological mapping, rock sampling and soil sampling (Singh, 2008).

9.2 2006 Program

In 2006, SMG expanded its exploration work by core drilling 21,886 m in 88 holes on both the Main and North Zones. In addition, 5,008 m of RC drilling in 50 holes were drilled along the eastern edge of the Main Zone; the South Zone; the Placer area west of the Main Zone; and the Cedar Creek area. Grid soil sampling (1,515 samples), and regional and property scale geological mapping were also completed. Rock samples, totalling 465, collected on a regional scale led to the discovery of the Oscar showing north of Spanish Creek. Geophysical work comprised an airborne electromagnetic and magnetic survey over the Property. Other airborne work included orthophotographs, from which were produced 1:1000 scale 0.3 m resolution orthophotos and topographic maps with precise 2-m contours (Singh, 2008).

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

The following year, 2007, SMG conducted 26,993 m of core drilling in 126 holes, focusing on infill drilling on the Main Zone for geological resource modeling, but also tested outlying areas (Singh, 2008). Limited geological mapping, soil sampling (450 samples) and rock sampling (127 samples) were also performed. Metallurgical testing involved the analysis of four composite samples by various flotation techniques to determine preliminary gold recoveries. In addition, a 30-person camp and core logging facility was built on SMG's private property located within the village of Likely.

9.4 2008 Program

A large drilling program consisting of 40,449 m of NQ and NQ2 core drilling in 161 holes was done in 2008 (Peatfield et al., 2009). Drilling focused 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.

Geological mapping was done in the Main Zone, primarily on newly exposed outcrop from pad building. Mapping was also done in the ROG and Cedar Creek areas. In total, 341 soil samples were collected between the Main Zone and the ROG area to the south. Environmental baseline studies were limited to monitoring weather stations.

9.5 2009 Program

In 2009, definition drilling continued in the Main Zone with a program of 62 core drillholes, totalling 13,769 m (AGP, 2010). 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 at a different lab from the NQ samples. In 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.

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 (Montgomery, 2009). Other work included reconnaissance geological mapping, rock sampling (41 rock samples) and preliminary re-interpretation of historic data. 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).

9.6 2010 Program

The 2010 exploration program consisted of 20 core drillholes within and peripheral to the Main and North Zones of the deposit, for a total of 6,834 m (Koffyberg, 2011). 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.

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

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 (11-DDH-986, 987, 988), 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 23.5 m of 0.58 g/t Au at a depth of 484.5 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.

Exploration work was also performed in the southeast part of the Property. A grid soil survey was performed, outlining a copper anomaly. 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. Other work included an airborne geophysical survey, which was carried out over the Property in late 2011. This involved a magnetic and DIGHEM V® electromagnetic airborne survey, which was carried out by Fugro Airborne Surveys Ltd. 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, 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 (Giroux and Koffyberg, 2012). This work finished on June 18, 2012. 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.

9.9 2013 Program

A review by Dr. M. Beattie, PEng, and CEO of SMG compared gold grade determinations of core drilling (2005 to 2012) versus RC drilling (2004 to 2005) (Appendix 9-1). 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 (<7m) 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.

9.10 2014 Program

In April of 2014, a Technical Report on an update mineral resource estimate was produced (Giroux and Koffyberg, 2014). Later that year, additional RC drilling was carried out on the Main and North Zones, totalling 2,621 m in 18 holes.

10.0 Drilling

SMG has been drilling on the Property since 2005. Table 10-1 summarizes the drilling activity on the deposit from 2005 onwards by SMG. See Appendix 14-1 for the list of drillholes used in the 2016 Resource Estimate.

Table 10-1 Summary of Exploration Drilling Activity by Spanish Mountain Gold

Year	Drill Type	No. of Holes	Metres	Core size
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

For the 2010, 2011 and 2012 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 and 2014 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 2009 to 2012 core drillholes are shown on Figure 10-1 to Figure 10-4, respectively. The 2013 and 2014 RC drillholes are shown on Figure 10-5 and Figure 10-6.

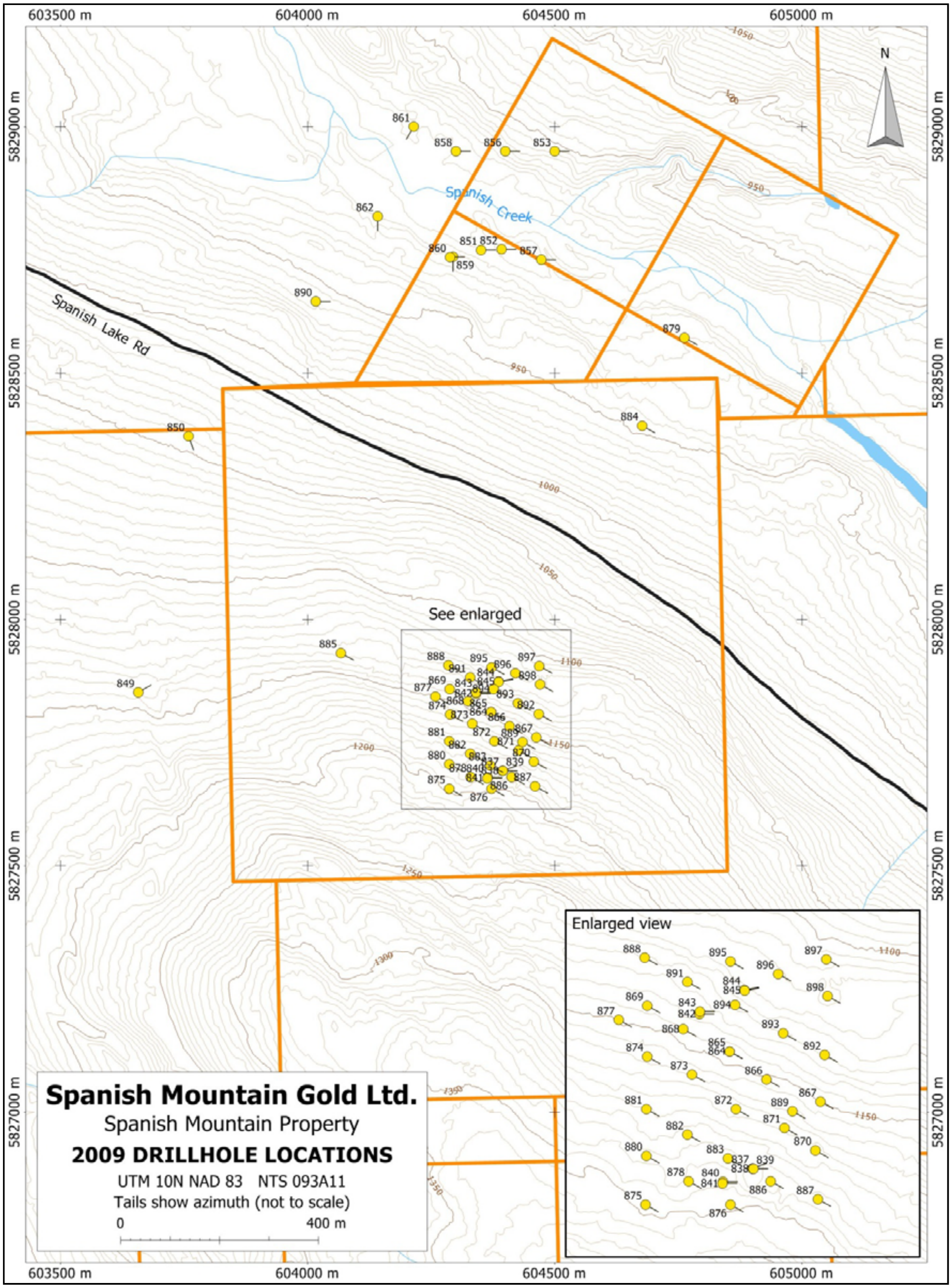


Figure 10-1 2009 Drillhole Locations

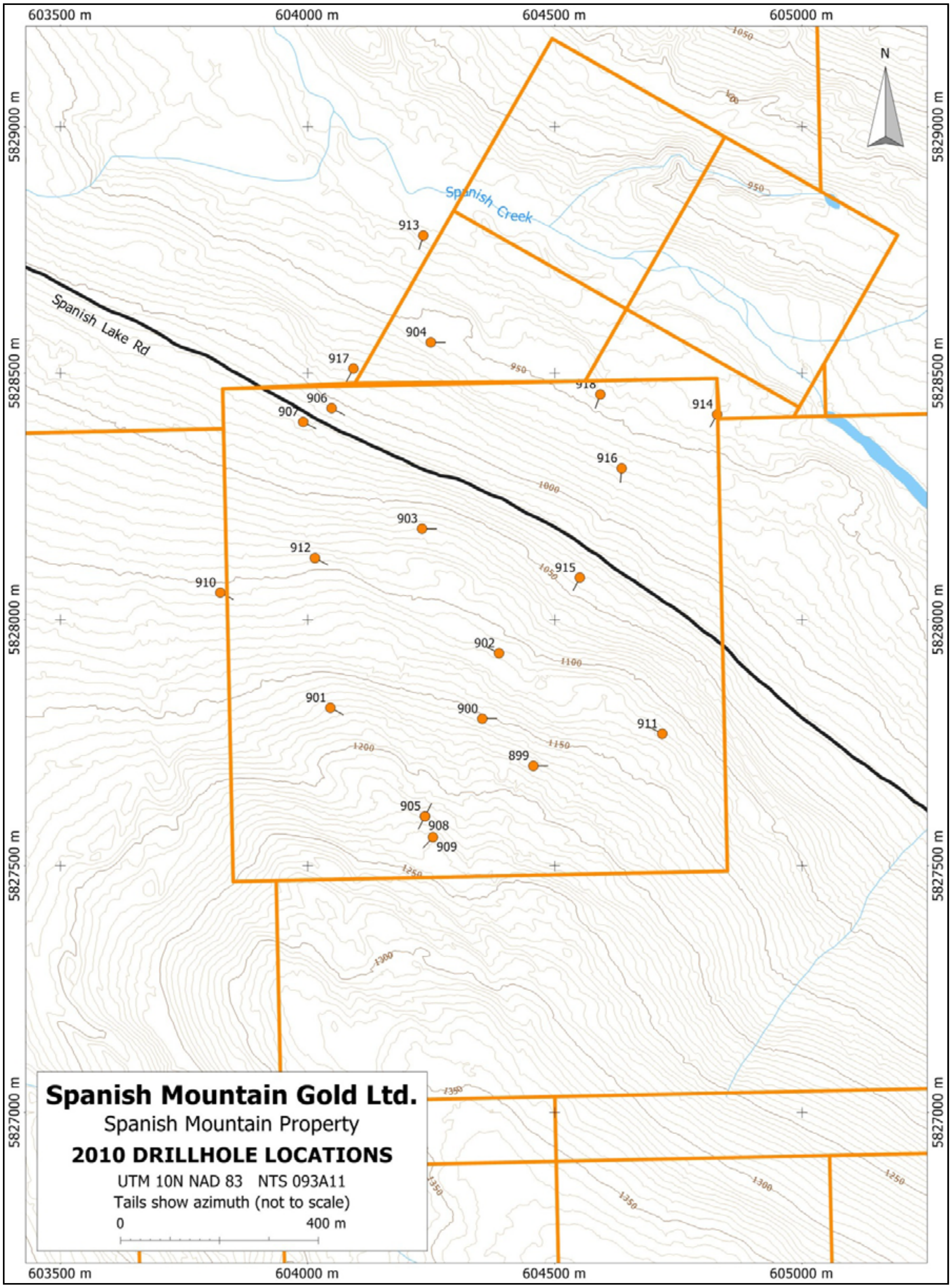


Figure 10-2 2010 Drillhole Locations

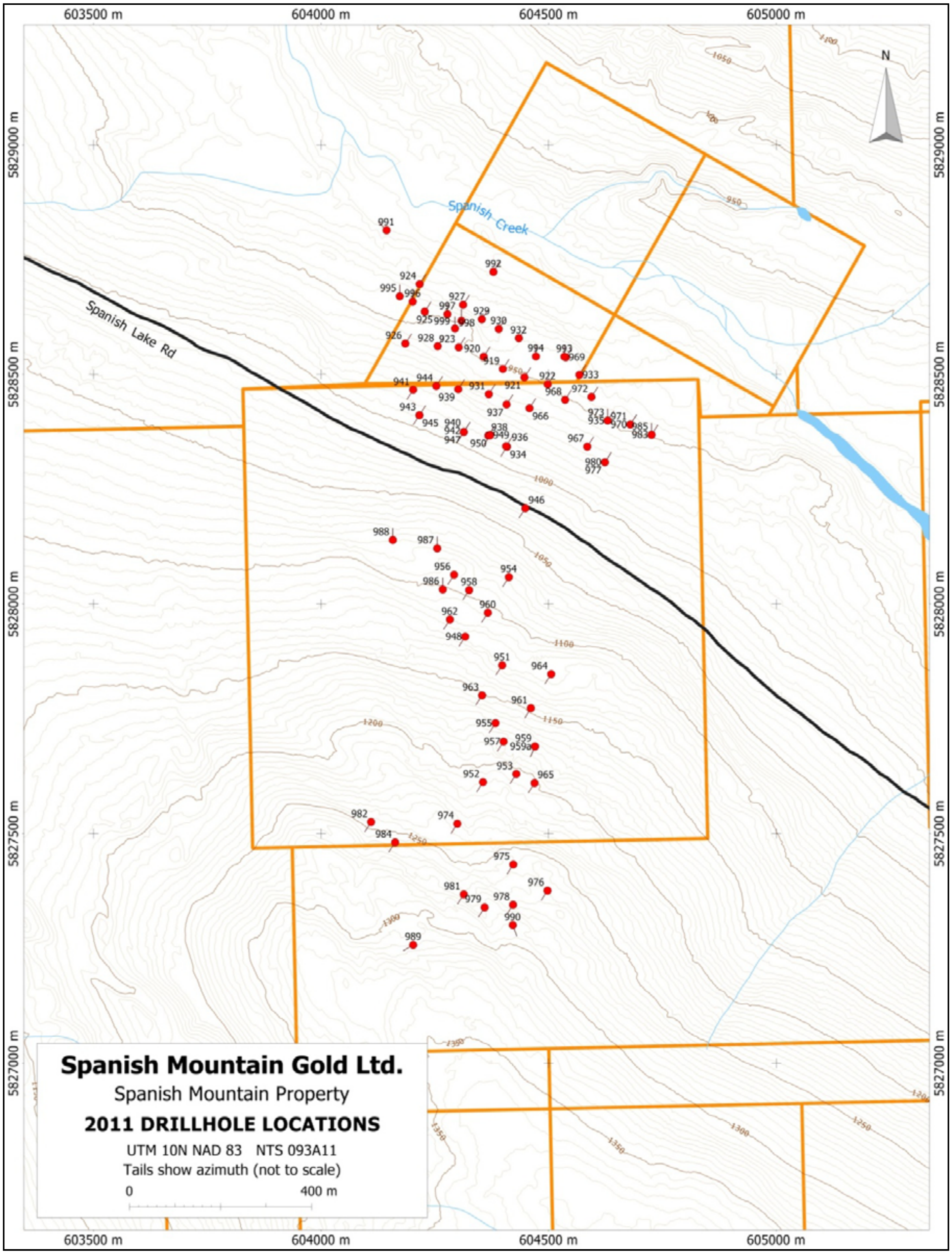


Figure 10-3 2011 Drillhole Locations

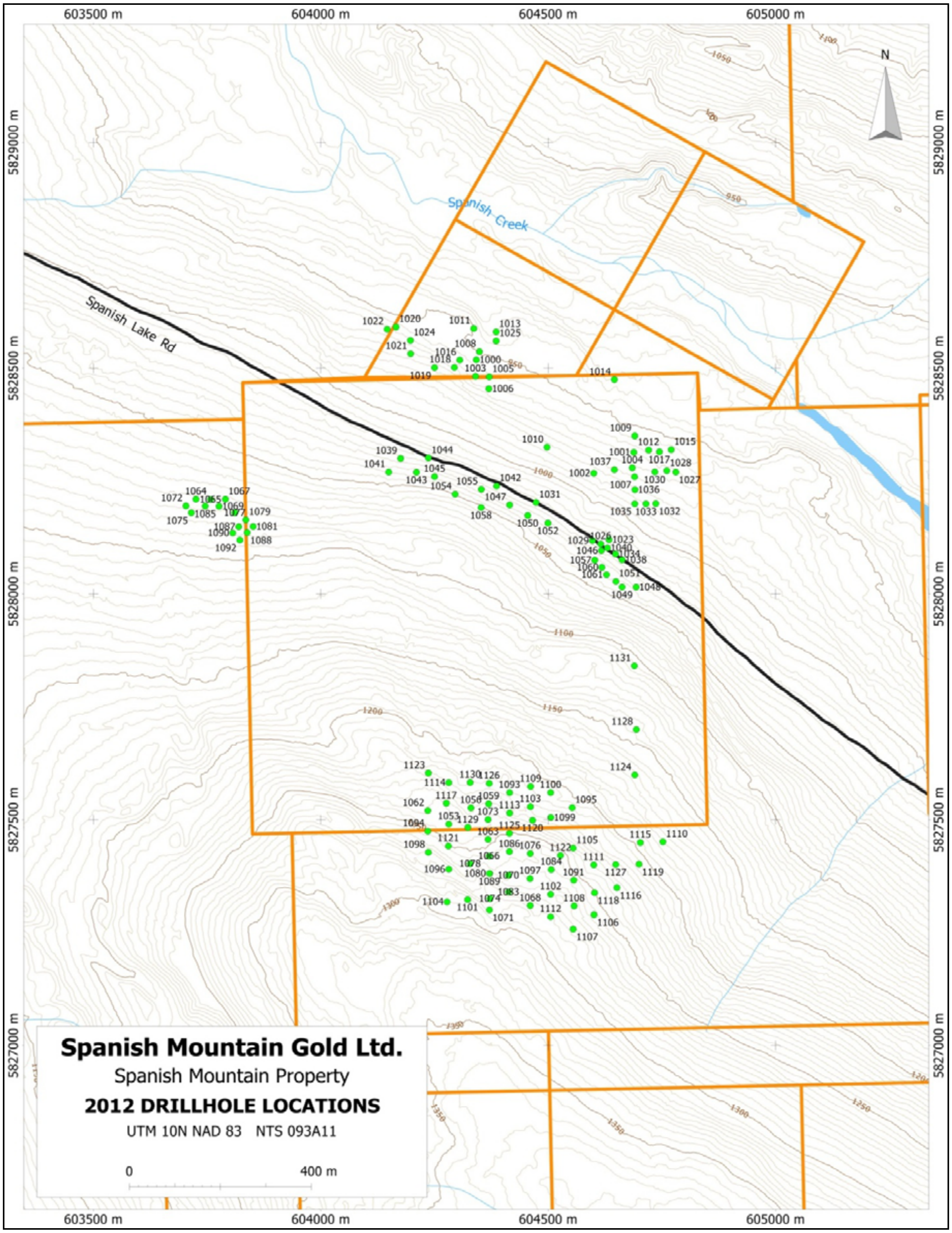


Figure 10-4 2012 Drillhole Locations

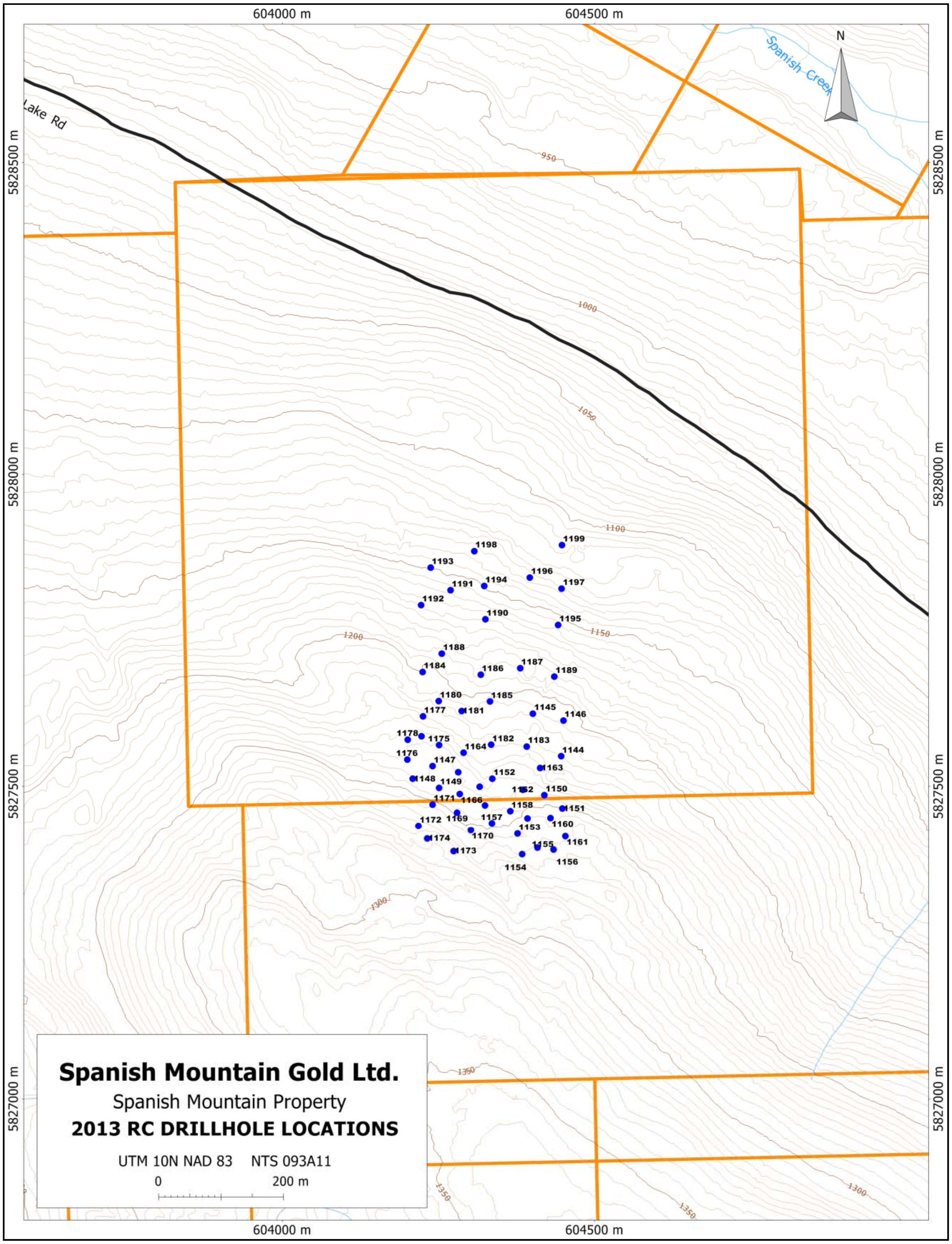


Figure 10-5 2013 Drillhole Locations

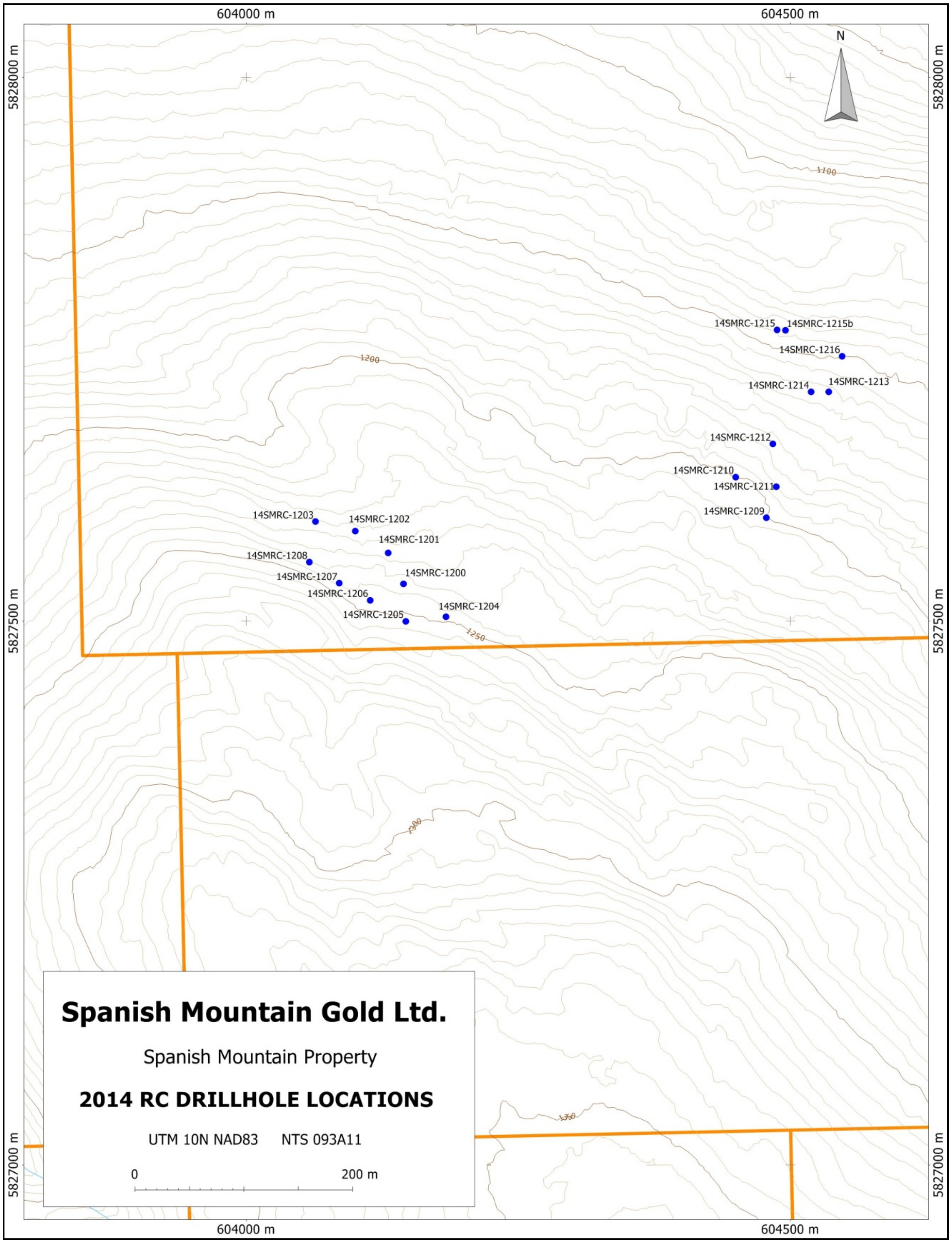


Figure 10-6 2014 Drillhole Locations

11.0 Sample Preparation, Analyses and Security

11.1 Sample Collection and Preparation

The following describes the sampling methods used by SMG in 2010, 2011 and 2012 core drilling programs and the 2013 RC drilling program. Sampling methods used during the 2004 to 2009 programs are described by Peatfield et al. (2009) and by AGP (2010). The information in this section was obtained from SMG, ALS Global Minerals Lab (ALS), and reports by co-author Gilmour, that summarize a Property visit on April 22, 2012, for the core drilling programs and a Property visit on August 23, 2013, for the RC drilling program.

Drill core and cuttings were transported by SMG personnel to SMG's core logging facility, where rock quality designation (RQD) procedures, core/cuttings logging, core splitting and core/cuttings 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 7km from the Main and North Zones. Core storage is also located there.

11.1.1 2010-2012 Core Drill Programs

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.

11.1.2 2013-2014 RC Drill Programs

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 they 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 chutes. One half of the material from the splitter was collected in a pre-labelled plastic sample bag; the other half was discarded. When a field duplicate was taken, the material from both sides of the riffle splitter was collected 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 each 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 <12kg. Samples weighing over 12kg 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 two to three minutes, at which time the water was decanted and the fines transferred into a labelled 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 two to three 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 two to three 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 binocular 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.

11.2 Sample Analyses

Analytical procedures used at ALS were:

- Gold: Fire assay gold, specifically the 1-kg screen metallic method (Au-SCR21), which uses both an atomic absorption finish and a gravimetric finish
- Multi-element: Four-acid multi-element analysis by ICP and MS (ME-ICP61)

The 1-kg screen metallic method involved crushing the entire sample in an oscillating steel jaw crusher for 70% to pass -10 mm. A 1-kg split was pulverized and passed through a 150 mesh (100 µm grain size), producing a plus fraction (i.e., >100µm) and minus fraction (i.e., <100µm). 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.

This ALS facility is certified to standards within ISO 9001:2008 and has received accreditation to ISO/IEC 17025:2005 from the Standards Council of Canada (SCC).

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. Sample shipping was done through a private trucking courier to ALS in North Vancouver, BC. The security procedures meet quality control standards.

11.4 Quality Control and Quality Assurance Program

Since December 2011, SMG has retained Discovery of Vernon, BC, to independently monitor the QC/QA procedures. The monitoring was done under the supervision of W. Gilmour, PGeo, 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 reject samples
- duplicate samples of core or of RC cuttings, 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.

Since QC/QA procedures have varied though the long period of drill exploration, specific QC/QA measurements are not available for all the data used in the Resource Estimate.

11.5 Contamination

The purpose of field blank samples was to check for contamination within the preparation (crushing, pulverizing) process. 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. For the 2011 program, the blanks were inserted randomly in the sample stream about every 30 samples.

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 almost every sample being less than the 0.05 g/t detection for metallic gold analysis – demonstrating no significant contamination during the sample preparation process.

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. 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 containing anomalous gold, when compared to the 2011, 2012 and 2014 drilling. However, these anomalous samples were generally not near mineralized zones, hence no significant contamination was noted. 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.

11.5.1 **Precision**

Duplicate 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 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

- **reject (prep) 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 (prep) 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 cutting, 2013	15	6	3

11.5.1.1 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

n = 202

Precision Values (%)				
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

n = 110

Precision Values (%)				
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.

11.5.1.2 Reject Duplicates

For the 2011 drilling used in the 2011 Resource Estimate, the laboratory systematically produced, every 30 samples (901 pairs), another sample from the saved reject (crushed) core. Sample pairs containing an average grade of at least 0.040 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

n = 418

Precision Values (%)				
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

n = 209

Precision Values (%)				
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 (reject or prep) 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 2013 RC Reject Duplicates - Precision Values

n = 106

Precision Values (%)				
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.

11.5.1.3 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 0.040 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

n = 7278

Precision Values (%)				
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 0.040 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

n = 4092

Precision Values (%)				
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.

11.5.2 Accuracy

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.

Standards have been analysed throughout the drill programs from 2005 to 2012. 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. Minor re-analysis of a group of samples was done. However, no changes in the results were warranted.

It is the opinion of the author Gilmour that 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, 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 R. Johnston, P.Geo., of Mincord Exploration Consultants. The 2005 core and RC drilling program by SMG was conducted under the supervision of R. Darney, P.Geo., of Pamicon.

The 2006 to 2009 drilling programs by SMG were completed under the direction of R. Singh, P.Geo., of Pamicon. G. Peatfield, P.Eng., reviewed the 2008 and 2009 work and agreed that the results were generally acceptable (Peatfield et al., 2009).

The 2010 core drill program was carried out by SMG under the supervision of S. Morris, P.Geo. of SMG. Drill core from the 2010 drill program has been examined on site, and drill logs and analytical certificates, along with QC/QA procedures, have been reviewed by A. Koffyberg, P.Geo., of Discovery Consultants.

The 2011 and 2012 core drill programs pertaining to the Resource Estimate were carried out by SMG under the supervision of J. Stoeterau, P.Geo., of SMG. Drill core from the 2011 and 2012 drill programs have been examined, and drill logs, and analytical certificates, have been reviewed by Koffyberg.

The 2013 and 2014 RC drill program was carried out by SMG under the supervision of J. Stoeterau, P.Geo., of SMG. Qualified Persons from Discovery monitored the drilling, sampling, QC/QA procedures, reviewing analytical certificates throughout the drill program. The co-author, Gilmour, was responsible for reviewing the results, including QC/QA.

13.0 Mineral Processing and Metallurgical Testing

Several metallurgical test work phases have been completed between 2007 and 2017 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

Flotation of the potential ore grade material at a coarse grind size readily produces a gold concentrate with high recoveries. A large portion of the minerals processing test work has focused on rejecting organic carbon to produce a leachable concentrate. 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
table continues…		

Document or Test Program	Author or Laboratory	Date
Metallurgical Testing on Samples from the Spanish Mountain Gold Project, Report No. KM2637	G&T	September 7, 2011
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 test work on samples collected from exploration drillholes has included the following:

- comminution characterization
- 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%, 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 value and TOC values ranged between 0.03 and 2.03% TOC and averaged 0.89% TOC.

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% 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.

13.4 Gravity Concentration

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

Gravity recovered concentrate generally was 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.

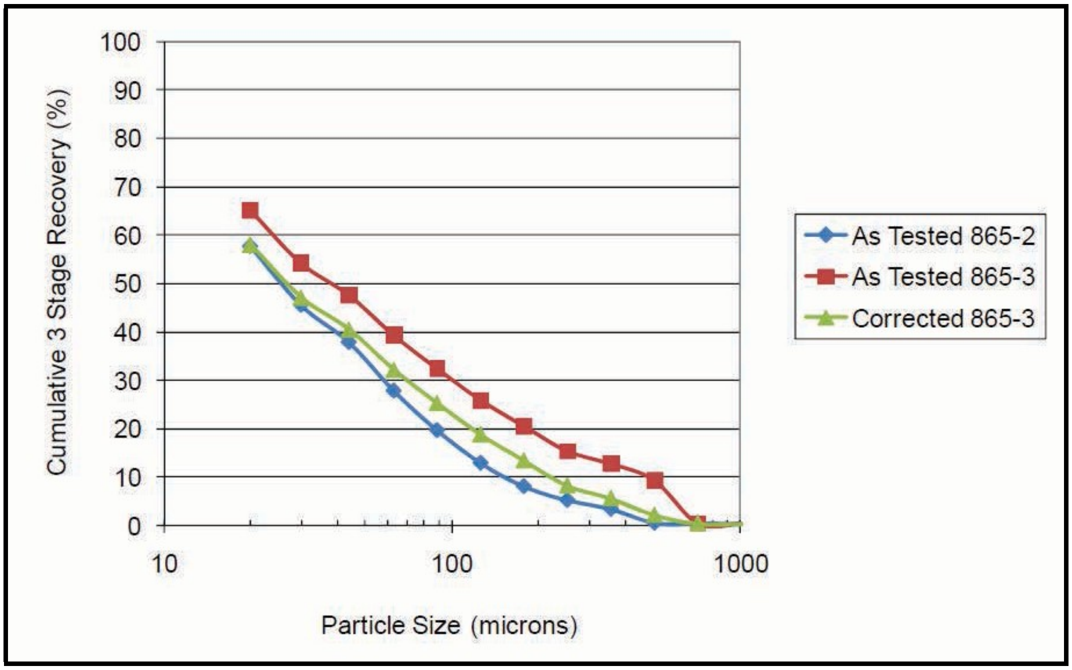


Figure 13-1 Cumulative Three-stage EGRG Results (Revised) – Knelson Research

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 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 for the leaching of the gold in a CIL circuit. 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.

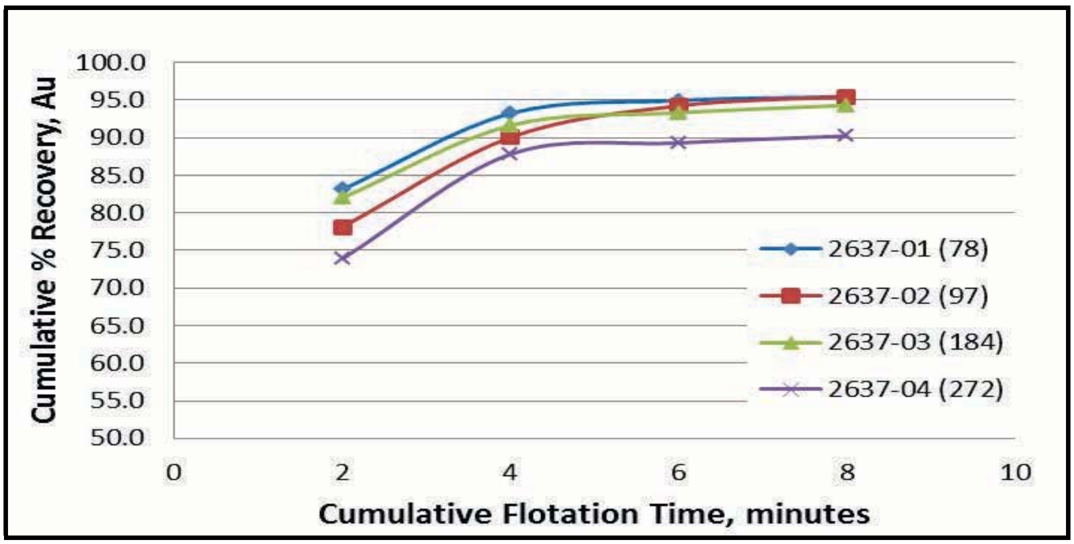


Figure 13-2 Primary Grind Size versus Flotation Kinetics – G&T

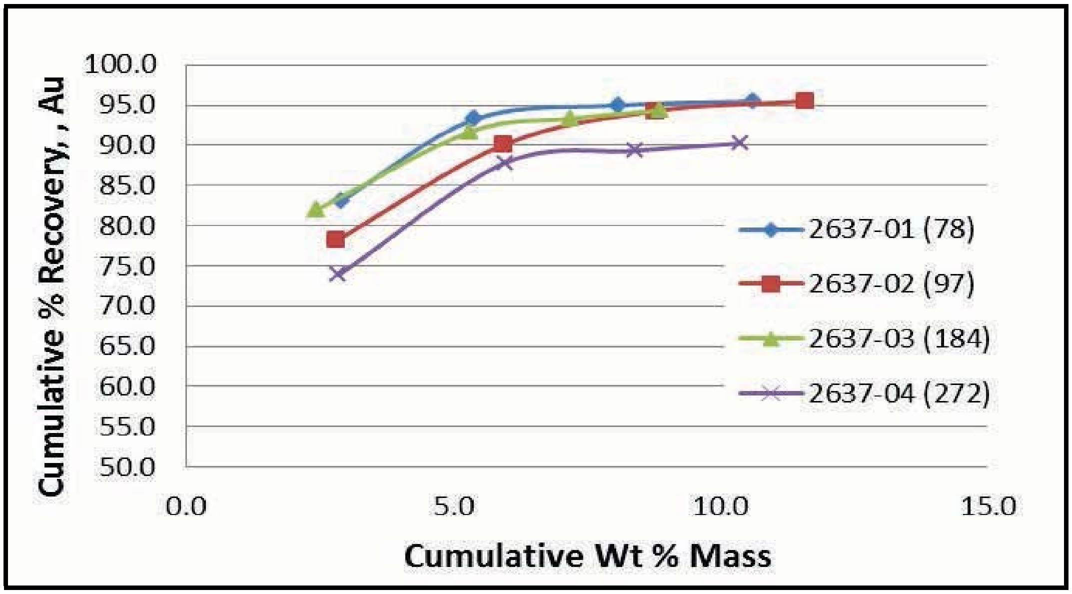


Figure 13-3 Primary Grind Size versus Mass Recovery – G&T

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. CMC has been used to depress the carbon (discussed in further detail in the following section). Reagent addition rates are not yet optimized.

13.5.3 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.

13.5.4 Regrind of Concentrate

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 13-4 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.

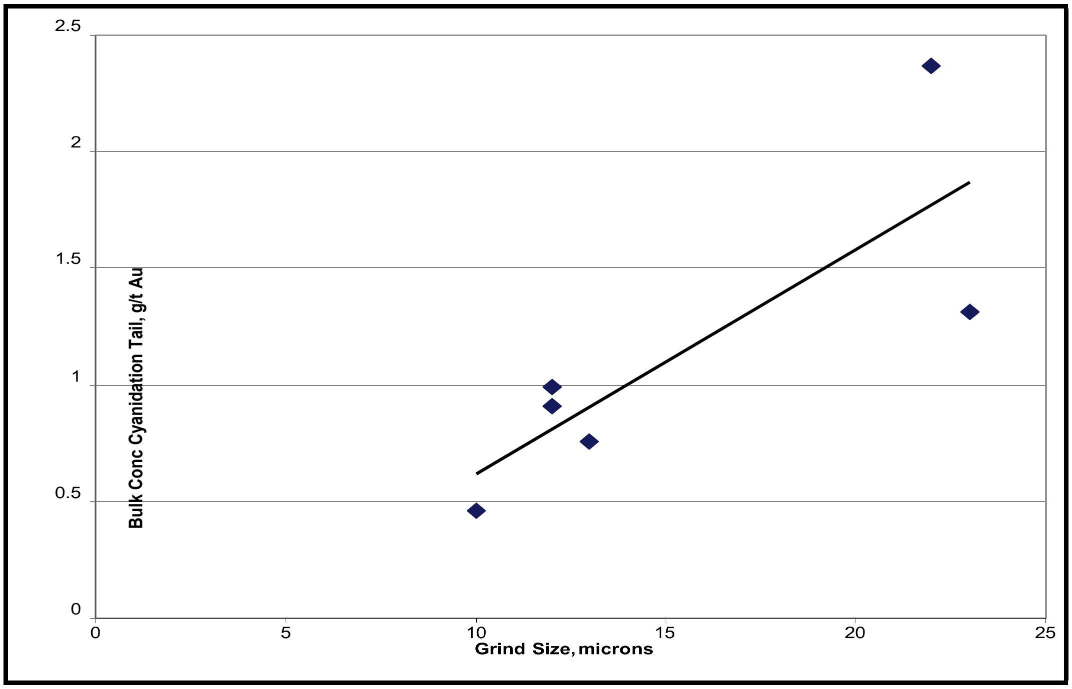


Figure 13-4 Flotation Concentrate Cyanidation Tailings versus Regrind Size – G&T

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. Results from the initial leach tests 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 13-5.

Table 13-5 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 13-6.

Table 13-6 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 13-7. Material for these leach tests was created from a sample which had been gravity processed, included 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 13-7 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.

13.6.3 Pre-Flotation Trade Off

Conceptual test work conducted at Metsolve in 2017 using a concentrate produced without pre-flotation, and including CMC treatment in cleaner flotation followed by regrinding to minus 20 microns achieved a leach gold recovery of 98% and a silver extraction of 90%, demonstrating that pre-flotation is not required.

13.7 Variability

Variability test work has been carried out at SGS and G&T to provide data for process design criteria, and test the variability among the samples using the elected processing method. Composite samples were collected from various drillholes located across the deposit. A review of the test work shows that optimization of flotation conditions is not yet complete and additional test work is warranted to assess variability with the simplified flowsheet.

13.8 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 can produce a concentrate with a low enough TOC content to overcome potential preg-robbing properties.

Gold overall process recovery for the PEA is estimated to be 89% from a process that includes grinding to a P_{80} of 184 microns, rougher flotation, flotation cleaning with CMC and subsequent CIL of reground concentrate.

Silver occurs in minor economic proportions at SMG. Metallurgical test work indicates an estimated 40% overall silver process recovery.

14.0 Mineral Resource Estimates

G. Giroux, PEng, of Giroux Consultants Ltd. was retained to produce an updated Resource Estimate ("Resource") on the Spanish Mountain Gold Deposit located approximately 6 km east of Likely, BC, and 70 km northeast of Williams Lake. The effective date for this Resource is October 3, 2016, the day the data was received.

Since the 2012 PEA report by Tetra Tech (Tetra Tech, 2012) an extensive study and test program comparing reverse circulation to core drilling results on the project has been completed. The test program of 56 reverse circulation drillholes in the Main Zone was completed in 2013. In 2014, an additional eighteen reverse circulation holes were completed within the main zone starter pit area.

G. Giroux is the qualified person responsible for the Resource Estimate. Mr. Giroux is a qualified person by virtue of education, experience and membership in a professional association. He is independent of both the issuer and the vendor applying all of the tests in Section 1.5 of National Instrument 43-101. Mr. Giroux has last visited the Property on June 29, 2011.

14.1 Data Analysis

In total, 890 drillholes were provided, but only 669 core and 145 reverse circulation drillholes penetrated the various geologic solids. This Resource is based on RC drillholes, including 74 infill holes completed since the previous Resource Estimate reported in the 2012 PEA Report (Tetra Tech, 2012). In total, 669 core drillholes (153,828 m) from 2005 to 2012 inclusive and 145 RC holes (19,034 m) from 2004 to 2006 and from 2013-2014 have been used in the Resource Estimate, for a grand total of 814 drillholes (172,862 m). A complete list of the drillholes is provided in Appendix 14-1. Missing or un-sampled intervals were filled with 0.001 g/t Au. Samples not sampled for silver from earlier drill campaigns and the last 18 RC holes were left blank.

14.1.1 Comparison of Core Drill Results to RC Drill Results

Initial comparisons between reverse circulation drilling (RC) and core drilling (DDH) results identified a bias between these two drill methods that was quantified in the 2011 Resource Estimate (Giroux and Koffyberg, 2011). Figure 14-1 taken from this report shows a comparison of the gold grade distributions for the two drilling methods. The RC results appeared to be biased high relative to the DDH [core] results within the same volume of rock. As a result, RC drillholes at Spanish Mountain had not been used in the 2011 and 2012 Resource Estimates.

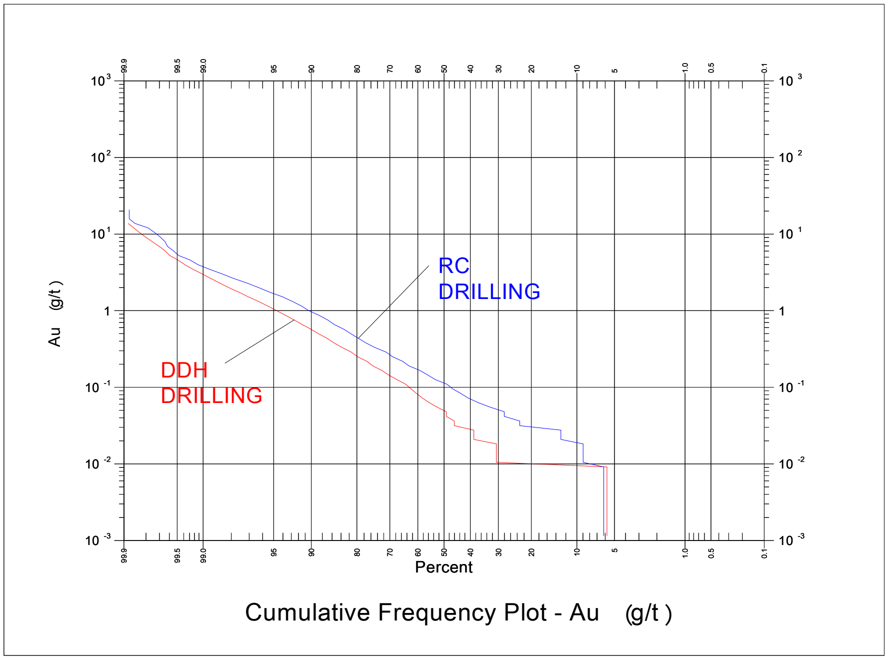


Figure 14-1 Lognormal Cumulative Frequency Plot for Gold from core (DDH) drilling (red) and reverse circulation drilling (blue)

Since the 2012 PEA report a study comparing core drill (DDH) results and reverse circulation (RC) results at Spanish Mountain as well as comparing entire core interval analyses with standard protocol samples was completed by M. Beattie. This report is included as Appendix 2. The study completed by Beattie and others came to a different conclusion:
"Available data have been critically analyzed to determine if the grade determination results for the Spanish Mountain Gold Project diamond drill samples that have been the basis for Resource Estimates to date have a bias. The data for the QA/QC programs followed to date, the results from the analysis of entire core intervals and the results of RC drilling are analyzed in this report. Based on the comparison of these various results with those obtained by diamond [core] drilling with the standard sample preparation protocol it is concluded that there is a negative bias to the existing data base and that the resource grade is understated to a material degree. For the purpose of this analysis a "material" increase in grade is considered to be one of at least 15%."

Based on the conclusions of this study SMG conducted a drill test in 2013, drilling 56 additional reverse circulation drillholes within a test area of the Main zone defined by the coordinates:

East – 604190 to 604460
North – 5827350 to 5828050
Elevation (masl) – 1300 to 950

To test the effects of using only RC holes, blocks within this test volume were re-kriged using only RC composites in a similar manner to the 2012 Resource Estimate. Within the RC Test Area only the Upper Argillite, Tuff and Lower Argillite domains were present and estimated. Since the RC holes did not penetrate below 950 elevation, only blocks above this level were compared. A comparison of block grades with the 2012 estimate, which used only core drillholes, is tabulated below.

Table 14-1 Comparison of average block gold grades between the Core (DDH) estimate and the RC Estimate within the Test Block

Domain	Drill type	Number of Blocks	Total Tonnage	Average Estimated Au (g/t)	% Increase
Upper Argillite	DDH holes	3,014	9,358,000	0.52	11.5%
	RC holes			0.58	
Tuff	DDH holes	8,498	26,670,000	0.40	25.0%
	RC holes			0.50	
Lower Argillite	DDH holes	23,939	74,330,000	0.24	41.7%
	RC holes			0.34	

Based on the conclusions and results from these two studies, all drillholes both RC and core (DDH) holes are used in the Resource.

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

All material, outside of these domains, was considered waste rock.

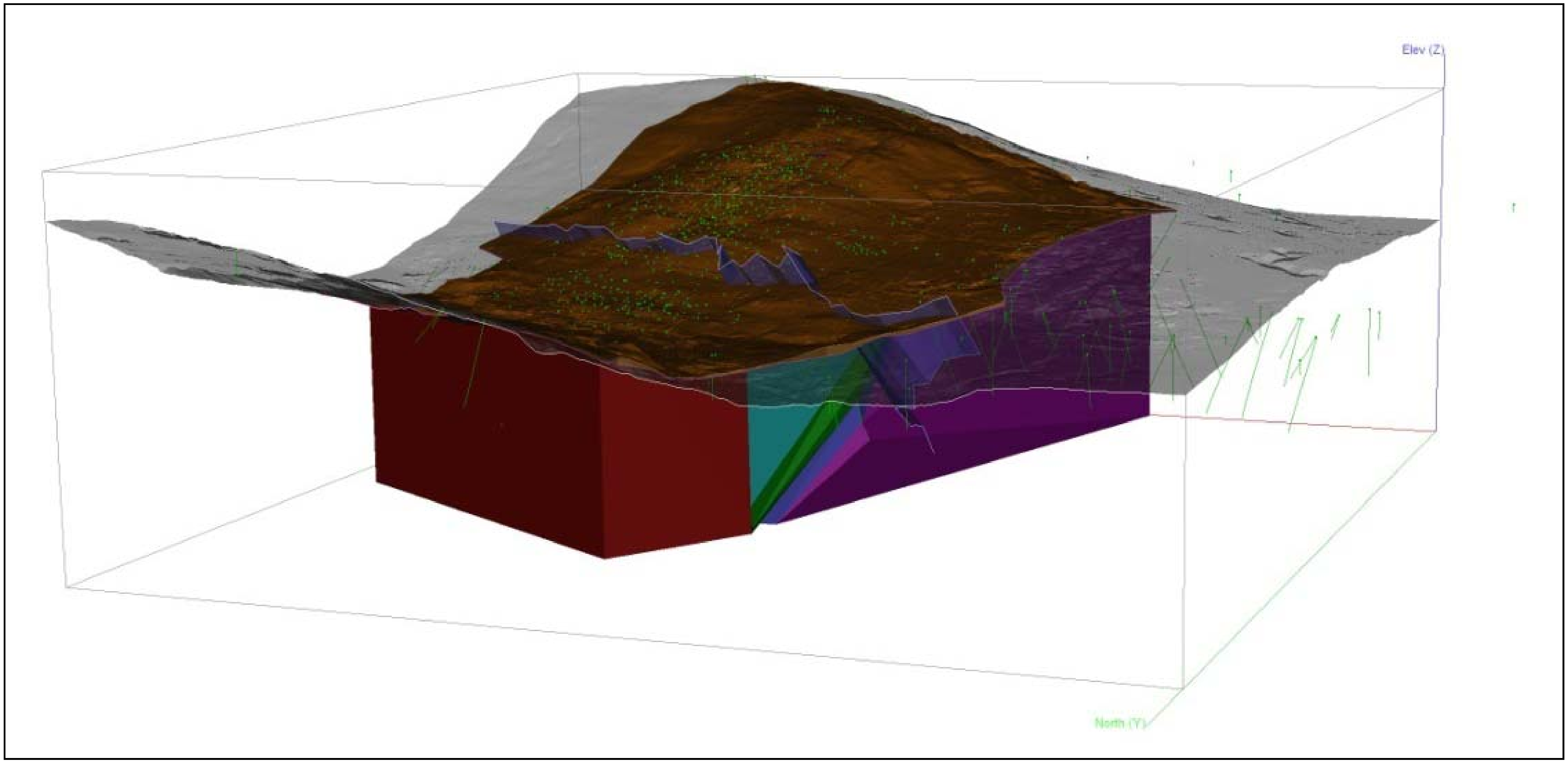


Figure 14-2 Isometric View Looking Southeast showing Lower Argillite in purple, Tuff in blue, Upper Argillite in green, Siltstone in blue green and North Zone Argillite in red. Inflection plane shown in blue, surface topography in grey and overburden in brown

Due to a significant change in dip of the Upper Argillite unit an inflection plane was used to separate the flatter dipping section from the steeper dipping section to the east.

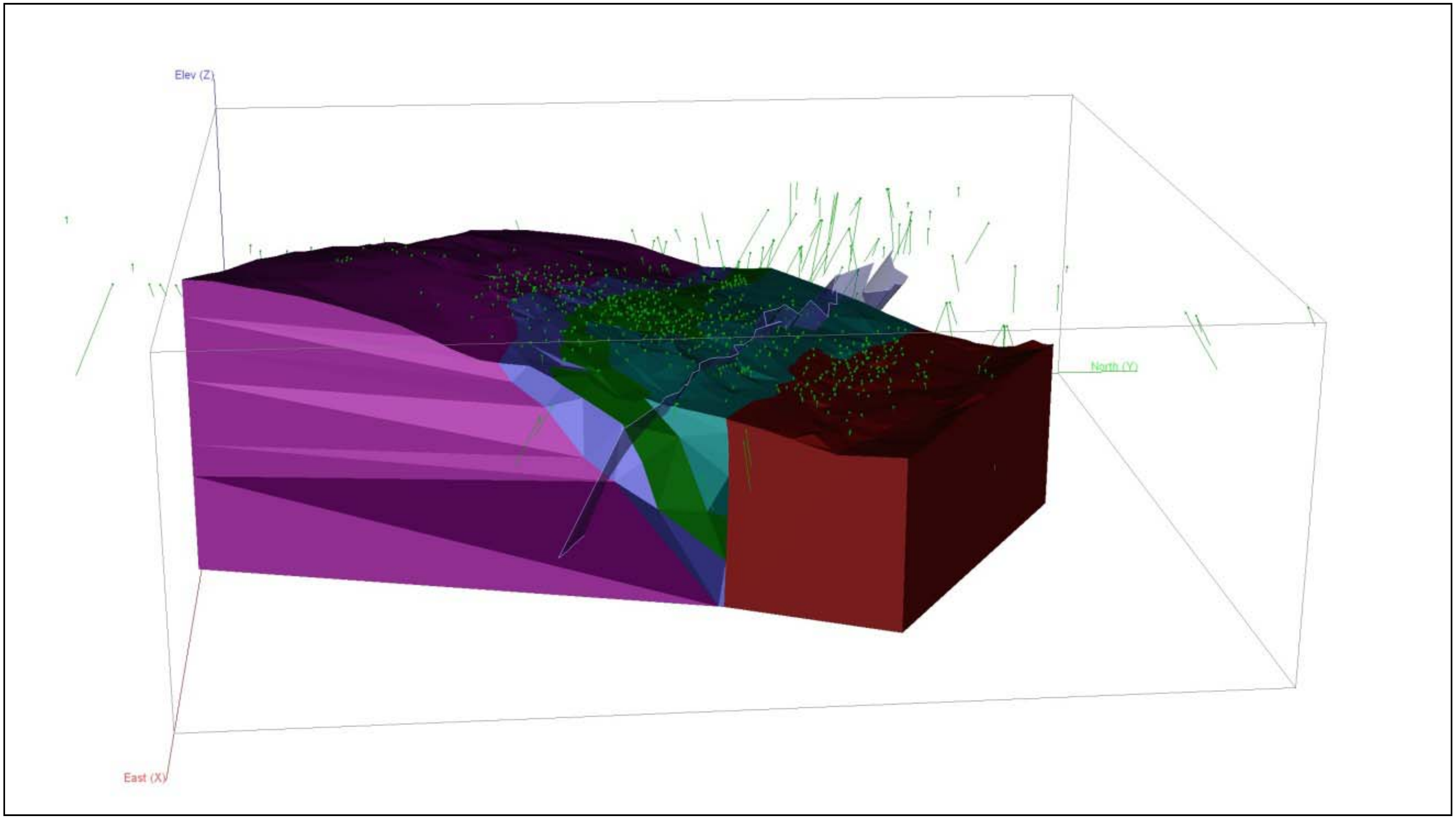


Figure 14-3 Isometric View Looking West showing Lower Argillite in purple, Tuff in blue, Upper Argillite in green, Siltstone in blue green and North Zone Argillite in red. The Inflection plane is shown in blue

The sample statistics for gold are tabulated below in Table 14-2 subdivided by the various geologic domains.

Table 14-2 Statistics for all Gold Assays in Geologic Domains

	Upper Argillite	Altered Siltstone	Tuff	Lower Argillite	North Zone Argillite	Waste
Number of Assays	14,737	9,850	22,136	40,890	17,289	7,882
Mean Au (g/t)	0.47	0.07	0.33	0.22	0.25	0.06
Standard Deviation	1.44	0.77	2.60	1.72	0.80	0.84
Minimum Value	0.001	0.001	0.001	0.001	0.001	0.001
Maximum Value	83.40	39.00	225.00	241.00	54.40	73.80
Coefficient of Variation	3.09	10.90	7.84	7.87	3.26	13.96

Table 14-3 Statistics for all Silver Assays in Geologic Domains

	Upper Argillite	Altered Siltstone	Tuff	Lower Argillite	North Zone Argillite	Waste
Number of Assays	13,359	9.560	19,481	34,386	17,037	7,410
Mean Ag (g/t)	0.86	0.41	0.44	0.59	0.66	0.64
Standard Deviation	1.287	0.67	1.20	0.76	1.36	1.05
Minimum Value	0.001	0.001	0.001	0.001	0.001	0.001
Maximum Value	88.90	28.20	84.10	30.00	103.00	23.00
Coefficient of Variation	1.47	1.64	2.72	1.29	2.08	1.63

The gold grade distributions within the mineralized domains were examined to determine if capping was required and if so at what level. In each case the distribution for gold was strongly skewed. A lognormal cumulative frequency plot was produced for gold in each domain and in all cases showed multiple overlapping lognormal populations. Capping levels were determined to reduce the effect of small high grade populations that can be considered erratic.

Table 14-4 Capping Levels for Gold Assays in Geologic Domains

Domain	Cap Level Au (g/t)	Number Capped	Cap Level Ag (g/t)	Number Capped
Upper Argillite	13.0	14	20.0	4
Tuff	30.0	15	30.0	4
Altered Siltstone	10.0	9	20.0	2
Lower Argillite	16.0	26	25.0	3
North Zone Argillites	15.0	5	30.0	5
Waste	2.0	5	10.0	5

None of the 2014 assays required capping.

The results from capping are shown below in Table 14-5.

Table 14-5 Statistics for Capped Gold Assays in Geologic Domains

	Upper Argillite	Altered Siltstone	Tuff	Lower Argillite	North Zone Argillite	Waste
Capped Au Assays						
Number of Assays	14,737	9,850	22,136	40,890	17,289	7,882
Mean Au (g/t)	0.47	0.07	0.33	0.22	0.24	0.05
Standard Deviation	1.44	0.77	2.60	1.72	0.59	0.12
Minimum Value	0.001	0.001	0.001	0.001	0.001	0.001
Maximum Value	83.40	39.00	225.00	241.00	15.00	2.00
Coefficient of Variation	3.09	10.90	7.84	7.87	2.43	2.35
Capped Ag Assays						
Number of Assays	13,359	9,560	19,481	34,386	17,037	7,410
Mean Ag (g/t)	0.86	0.41	0.44	0.59	0.65	0.64
Standard Deviation	0.99	0.62	0.97	0.76	1.02	1.00
Minimum Value	0.001	0.001	0.001	0.001	0.001	0.001
Maximum Value	20.00	20.00	30.00	25.00	30.00	10.00
Coefficient of Variation	1.15	1.52	2.23	1.27	1.57	1.57

14.2 Composites

The drillholes were "passed through" the mineralized solids with the point at which each drillhole entered and left the solid recorded. Uniform 2.5 m downhole composites were then produced to honour these mineralized boundaries. Intervals less than 1.25 m at the solid boundaries were combined with adjoining intervals to produce a uniform support of 2.5 ± 1.25 m. The statistics for 2.5 m composites are shown below.

Table 14-6 Statistics for 2.5 m Gold Composites

	Upper Argillite	Altered Siltstone	Tuff	Lower Argillite	North Zone Argillite	Waste
2.5 m Gold Composites						
Number of Composites	9,187	6,082	12,796	22,864	10,978	3,373
Mean Au (g/t)	0.44	0.06	0.28	0.19	0.24	0.048
Standard Deviation	0.72	0.26	0.87	0.52	0.42	0.093
Minimum Value	0.001	0.001	0.001	0.001	0.001	0.001
Maximum Value	12.34	6.16	24.58	14.72	9.03	1.29
Coefficient of Variation	1.63	4.53	3.17	2.74	1.79	1.96
2.5 m Silver Composites						
Number of Composites	8,421	6,018	12,264	21,673	10,928	3,373
Mean Ag (g/t)	0.86	0.40	0.43	0.59	0.65	0.59
Standard Deviation	0.92	0.50	0.69	0.63	0.82	0.89
Minimum Value	0.001	0.001	0.001	0.001	0.001	0.001
Maximum Value	20.00	12.4	26.14	13.74	23.39	8.44
Coefficient of Variation	1.07	1.25	1.62	1.06	1.26	1.51

The gold grade relationships between the various lithologies across contacts were explored using contact plots. Figure 14-4 shows a contact plot for gold in the Upper Argillite compared with the Tuff domain. The dashed vertical line represents the contact between these two units and the average grade for gold is show on both sides for samples extending away from this contact. It is clear that there is a sharp grade change going across this contact and as a result, there should be a hard boundary for grade estimation. A hard boundary means samples on one side are not used to estimate blocks on the other side.

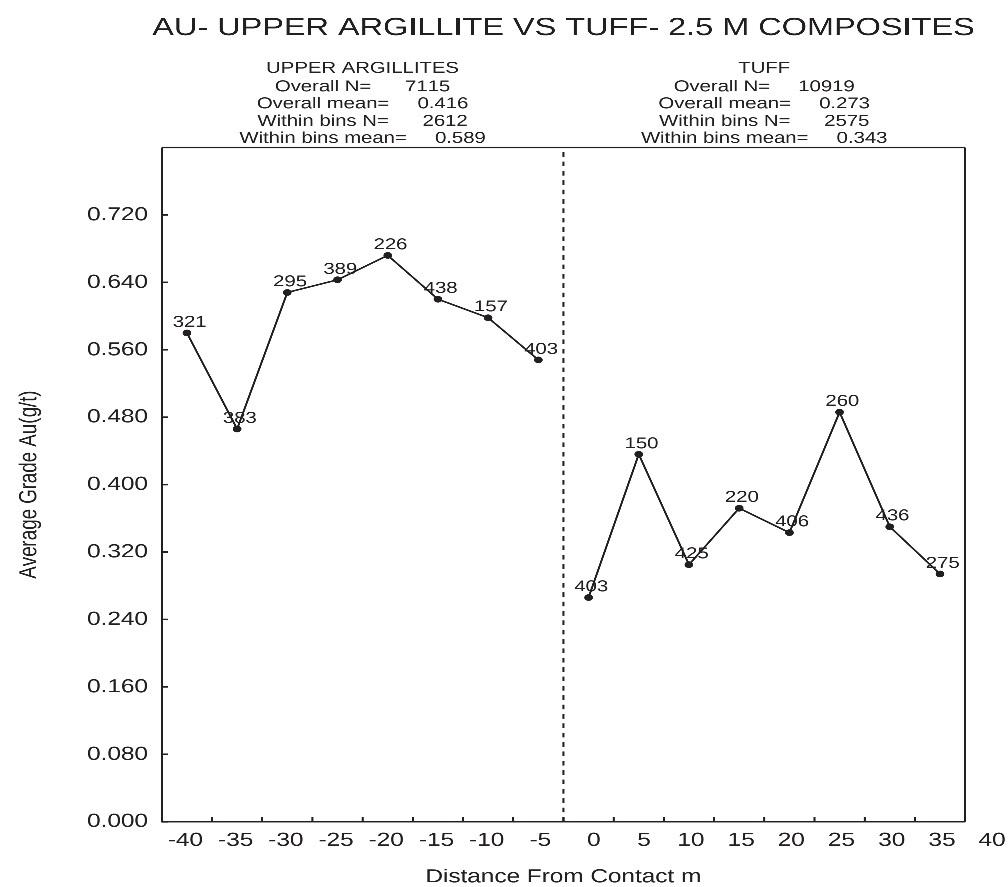


Figure 14-4 Contact Plot for Gold in Upper Argillite vs. Tuff Domain

A similar plot for Upper Argillite and Altered Siltstone shows a similar sharp contact across the contact (Figure 14-5) and again a hard boundary should be imposed for grade estimation.

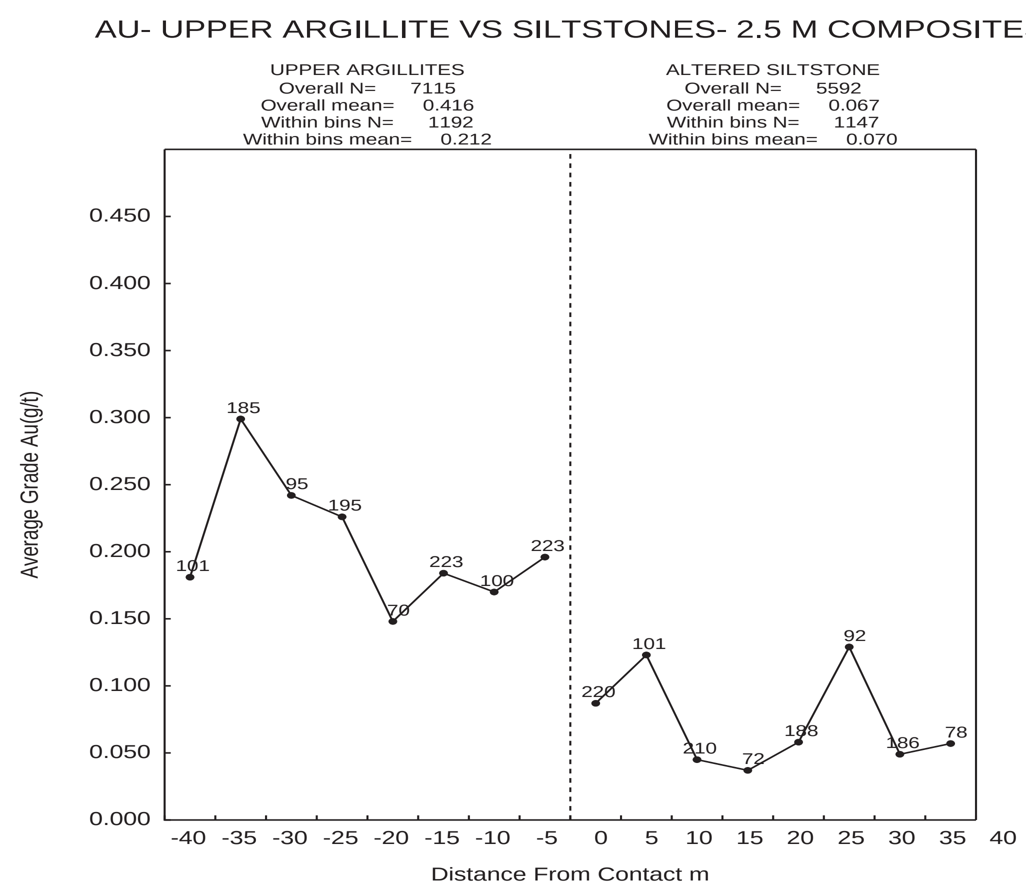


Figure 14-5 **Contact plot for Gold in Upper Argillite vs. Altered Siltstone Domain**

A contact plot for gold between the Lower Argillite and Tuff domain showed no significant changes across the contact (Figure 14-6) and these domains could be estimated with a soft boundary meaning composites from either could be used during estimation of blocks near this contact. The Upper and Lower Argillites and the Lower Argillite and Altered Siltstone do not contact each other.

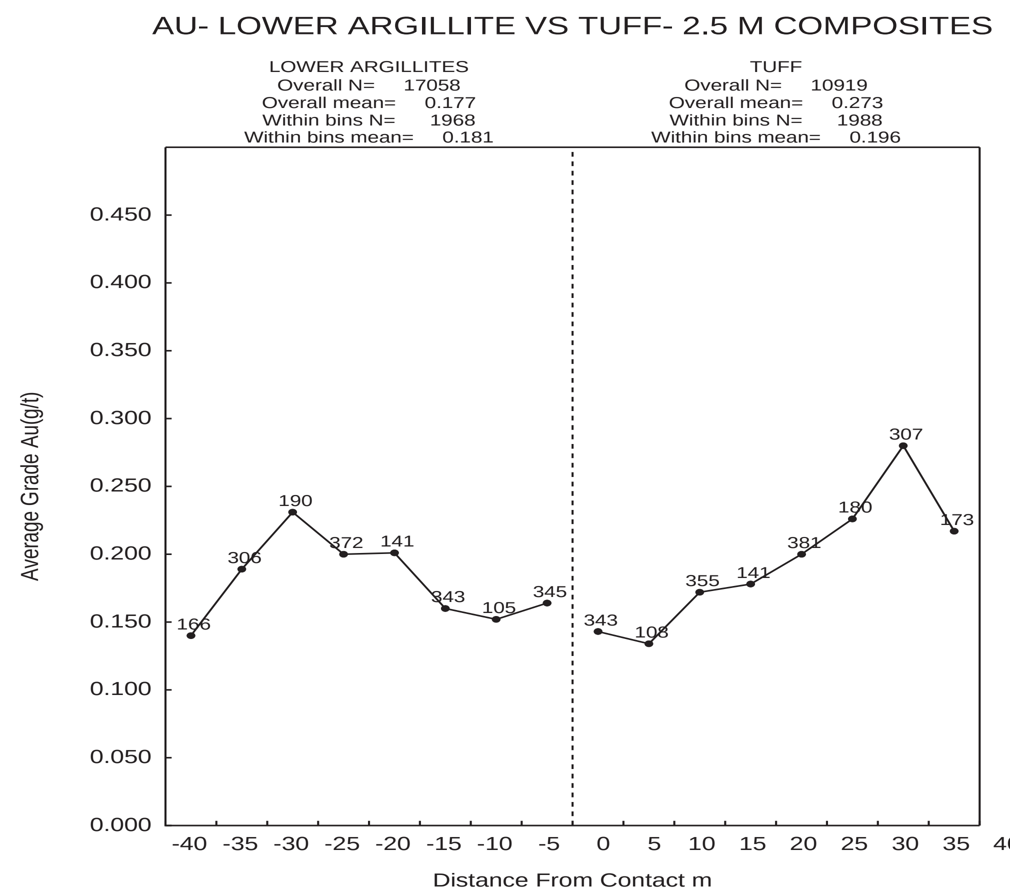


Figure 14-6 Contact Plot for Gold in Lower Argillite vs. Tuff Domain

14.3 Variography

Gold at Spanish Mountain was modelled separately for each geologic domain using pairwise relative semivariograms. In each case, semivariograms were produced in numerous directions within the horizontal plane. For each domain, the direction with the longest continuity was determined. The vertical plane perpendicular to this direction was then tested to determine the direction and dip of the longest continuity, with the third direction being orthogonal to this direction.

The model parameters are shown below.

Table 14-7 Summary of Semivariogram Parameters for Gold

Azimuth	Dip	C_o	C_1	C_2	Short Range(m)	Long Range(m)
Gold in Upper Argillite						
130°	0°	0.25	0.25	0.30	12	90
040°	-43°	0.25	0.25	0.30	18	82
220°	-47°	0.25	0.25	0.30	12	48
Gold in Tuff						
063°	0°	0.30	0.46	0.19	10	136
333°	-58°	0.30	0.46	0.19	12	100
153°	-32°	0.30	0.46	0.19	10	50
Gold in Lower Argillites						
130°	0°	0.20	0.30	0.29	8	80
040°	-15°	0.20	0.30	0.29	5	22
220°	-75°	0.20	0.30	0.29	12	110
Gold in Altered Siltstone						
140°	0°	0.20	0.12	0.20	10	64
050°	0°	0.20	0.12	0.20	15	40
000°	-90°	0.20	0.12	0.20	20	100
Gold in North Zone Argillite						
133°	0°	0.25	0.30	0.25	15	90
223°	-65°	0.25	0.30	0.25	12	80
43°	-25°	0.25	0.30	0.25	15	40
Gold in Waste						
Omni Directional		0.10	0.25	0.25	30	100

Table 14-8 Summary of Semivariogram Parameters for Silver

Azimuth	Dip	C_o	C_1	C_2	Short Range(m)	Long Range(m)
Silver in Upper Argillite						
130°	0°	0.16	0.10	0.26	12	90
040°	-43°	0.16	0.10	0.26	12	50
220°	-47°	0.16	0.10	0.26	22	40
Silver in Tuff						
063°	0°	0.10	0.10	0.21	15	36
333°	0°	0.10	0.10	0.21	10	20
0°	-90°	0.10	0.10	0.21	15	100
Silver in Lower Argillites						
130°	0°	0.10	0.10	0.20	20	80
040°	-15°	0.10	0.10	0.20	15	40
220°	-75°	0.10	0.10	0.20	15	120
Silver in Altered Siltstone						
140°	0°	0.14	0.08	0.12	20	120
050°	0°	0.14	0.08	0.12	15	60
000°	-90°	0.14	0.08	0.12	15	60
Silver in North Zone Argillite						
133°	0°	0.15	0.10	0.11	20	90
223°	-65°	0.15	0.10	0.11	30	100
43°	-25°	0.15	0.10	0.11	30	80
Silver in Waste						
Omni Directional		0.10	0.20	0.20	30	80

14.4 Block Model

A block model with blocks 15 x 15 x 5 m in dimension was superimposed over the mineralized geologic solids. The percentage of each block below surface topography, below overburden and within each mineralized solid was recorded. The block model origin is as follows:

Lower Left Corner

603125 E	Column size = 15 m	150 columns
5826305 N	Row size = 15 m	217 rows

Top of Model

1450 masl	Level size = 5 m	241 levels

No Rotation.

14.5 Bulk Density

In total, 2,155 measurements for specific gravity were taken using the weight in air – weight in water method. Samples were from drill core in holes 05-DDH-251 to 10-DDH-918 spread across the mineralized zone in all lithologies. The tables below summarize the results sorted first by lithology and then by gold grade. While there are slight differences in the various lithologies, there appears to be no correlation between specific gravity and gold grade. As a result, blocks within the block model were assigned a specific gravity based on lithology. A bulk density of 2.3 was assumed for overburden. Blocks straddling two or more lithologies were assigned a weighted average specific gravity.

Table 14-9 **Summary of Measured Specific Gravities sorted by Lithology**

Zone	Number of SG's	Minimum	Maximum	Average
Upper Argillite	305	2.39	3.00	2.76
Tuff	382	2.46	3.02	2.79
Siltstones	443	2.42	3.30	2.78
Lower Argillite	625	2.50	3.11	2.76
North Zone Argillite	392	2.60	3.28	2.77
Waste	8	2.66	2.92	2.80
Total	**2,155**	**2.39**	**3.30**	**2.77**

Table 14-10 **Summary of Measured Specific Gravities sorted by Gold Grade**

Au Grade Range	Number of SG's	Minimum	Maximum	Average
0.0 < 0.10	1,456	2.42	3.30	2.77
≥ 0.10 > 0.25	308	2.56	3.28	2.77
≥ 0.25 > 0.50	149	2.63	2.96	2.77
≥ 0.50 > 0.75	58	2.60	3.11	2.80
≥ 0.75> 1.00	43	2.70	3.00	2.79
≥ 1.00> 5.00	133	2.39	3.11	2.78
≥ 5.00	8	2.70	2.90	2.78
Total	**2,155**	**2.39**	**3.30**	**2.77**

14.6 Grade Interpolation

For this update, Ordinary Kriging was used to interpolate grades for gold and silver into blocks with some proportion within the mineralized solids. ARD minerals were not estimated. In all cases the kriging exercise was completed in a series of four passes with the search ellipse for each pass being a function of the semivariogram ranges.

Grades for gold and silver were estimated into blocks containing some percentage of Upper Argillites using only composites from Upper Argillites. A similar hard boundary strategy was used for blocks containing some percentage of Siltstones and North Zone Argillites. For blocks containing some percentage of Tuffs or Lower Argillites the search ellipse was allowed to see samples from either domain (a soft boundary). Within the Upper Argillites there was a pronounced change in bedding dip which was modelled by an inflection plane (see Figure 14-3). For blocks on the north side of this plane the search ellipse was steepened to find the required composites.

In all cases the first pass at estimation used a search ellipse with dimensions equal to one quarter of the semivariogram range in the three principal directions. A minimum of 4 composites were required to estimate the block. For blocks not estimated in pass 1 a second pass using one half the semivariogram ranges was completed. A third pass using the full semivariogram range and a fourth using twice the range completed the exercise. In all cases the maximum number of composites used was restricted to twelve with a maximum of three from any single drillhole allowed. In cases where a block containing two domains was estimated for one but not the other, a fifth pass was run to produce a grade for the unestimated domain.

In blocks containing more than one mineralized domain a weighted average for gold and silver was produced. For all estimated blocks on the edges of solids, with some percentage present of material outside the solid, a waste rock grade for gold was estimated using composites outside the mineralized solids. For every estimated block in the model a mineralized grade for gold and silver was produced as the weighted average of all mineralized domains and then a total block grade was produced by weighting in a zero grade for overburden and a grade for the contained waste rock. The kriging parameters for gold are tabulated below.

Table 14-11 Kriging Parameters for Gold in all Domains

Domain	Pass	Number Estimated	Az/Dip	Dist. (m)	Az/Dip	Dist. (m)	Az/Dip	Dist. (m)
Upper Argillite South of Inflection Plane	1	4,492	130/0	22.5	40/-43	20.5	220/-47	12.0
	2	13,455	130/0	45.0	40/-43	41.0	220/-47	24.0
	3	9,418	130/0	90.0	40/-43	82.0	220/-47	48.0
	4	4,833	130/0	180.0	40/-43	164.0	220/-47	96.0
Upper Argillite North of Inflection Plane	1	715	130/0	22.5	40/-70	20.5	220/-20	12.0
	2	4,937	130/0	45.0	40/-70	41.0	220/-20	24.0
	3	12,291	130/0	90.0	40/-70	82.0	220/-20	48.0
	4	24,900	130/0	180.0	40/-70	164.0	220/-20	96.0
Tuff	1	17,381	63/0	34.0	333/-58	25.0	153/-32	12.5
	2	40,666	63/0	68.0	333/-58	50.0	153/-32	25.0
	3	30,227	63/0	136.0	333/-58	100.0	153/-32	50.0
	4	32,515	63/0	272.0	333/-58	200.0	153/-32	100.0
Siltstones	1	811	140/0	16.0	50/0	10.0	0/-90	25.0
	2	6,855	140/0	32.0	50/0	20.0	0/-90	50.0
	3	29,591	140/0	64.0	50/0	40.0	0/-90	100.0
	4	31,108	140/0	128.0	50/0	80.0	0/-90	200.0
Lower Argillite	1	3,555	130/0	17.5	40/-15	3.75	220/-75	27.5
	2	28,768	130/0	35.0	40/-15	7.5	220/-75	55.0
	3	104,948	130/0	70.0	40/-15	15.0	220/-75	110.0
	4	230,102	130/0	140.0	40/-15	30.0	220/-75	220.0
North Zone Argillites	1	3,069	133/0	20.5	43/-25	10.0	223/-65	18.0
	2	21,802	133/0	41.0	43/-25	20.0	223/-65	36.0
	3	43,418	133/0	82.0	43/-25	40.0	223/-65	72.0
	4	100,468	133/0	164.0	43/-25	80.0	223/-65	144.0

14.7 Classification

Geologic continuity has been established on this Property by surface mapping and drillhole interpretation. This has led to the geologic domains that constrain the mineral estimate. Grade continuity can be quantified by the use of semivariograms with different ranges produce in different directions that relate to mineral deposition.

For this Resource Estimate, in general blocks estimated during Pass 1 using a search ellipse with dimensions equal to one quarter of the semivariogram range were classified as Measured. After this initial classification, the model was assessed and isolated blocks classified as Measured were reclassified as Indicated. Blocks estimated during Pass 2, using one half the semivariogram ranges, were classified as Indicated. All other blocks were classified as Inferred.

The results are tabulated below for the various classifications. A gold cut-off grade of 0.15 g/t has been highlighted based on an economic assessment, explained in Section 16.4, as a possible open pit cut-off.

Table 14-12 Spanish Mountain Measured Resource

Au Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade > Cut-off		Contained Metal	
		Au (g/t)	Ag (g/t)	Oz. Gold	Oz. Silver
0.10	53,670,000	0.47	0.64	800,000	1,110,000
0.15	**45,730,000**	**0.53**	**0.66**	**770,000**	**970,000**
0.20	38,470,000	0.59	0.66	730,000	810,000
0.25	32,530,000	0.66	0.65	690,000	680,000
0.30	27,840,000	0.72	0.64	650,000	570,000
0.40	20,750,000	0.85	0.64	570,000	430,000
0.50	15,740,000	0.98	0.65	500,000	330,000
0.60	12,120,000	1.11	0.65	430,000	250,000
0.70	9,600,000	1.23	0.66	380,000	200,000
0.80	7,710,000	1.35	0.68	330,000	170,000
0.90	6,290,000	1.46	0.69	300,000	140,000
1.00	5,120,000	1.58	0.70	260,000	120,000

Table 14-13 Spanish Mountain Indicated Resource

Au Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade > Cut-off		Contained Metal	
		Au (g/t)	Ag (g/t)	Oz. Gold	Oz. Silver
0.10	342,810,000	0.31	0.65	3,440,000	7,140,000
0.15	**260,800,000**	**0.37**	**0.67**	**3,110,000**	**5,650,000**
0.20	200,370,000	0.43	0.69	2,780,000	4,450,000
0.25	154,710,000	0.49	0.70	2,450,000	3,470,000
0.30	121,410,000	0.55	0.70	2,160,000	2,730,000
0.40	75,280,000	0.68	0.70	1,650,000	1,700,000
0.50	49,310,000	0.80	0.71	1,270,000	1,120,000
0.60	33,700,000	0.92	0.71	1,000,000	770,000
0.70	23,680,000	1.04	0.72	790,000	550,000
0.80	17,040,000	1.16	0.73	630,000	400,000
0.90	12,310,000	1.28	0.73	510,000	290,000
1.00	9,090,000	1.39	0.73	410,000	210,000

Table 14-14 Spanish Mountain Measured plus Indicated Resource

Au Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade > Cut-off		Contained Metal	
		Au (g/t)	Ag (g/t)	Oz. Gold	Oz. Silver
0.10	396,480,000	0.33	0.61	4,240,000	7,790,000
0.15	**306,530,000**	**0.39**	**0.64**	**3,880,000**	**6,280,000**
0.20	238,840,000	0.46	0.66	3,510,000	5,030,000
0.25	187,240,000	0.52	0.67	3,140,000	4,020,000
0.30	149,260,000	0.59	0.68	2,810,000	3,240,000
0.40	96,030,000	0.72	0.69	2,210,000	2,130,000
0.50	65,040,000	0.85	0.69	1,770,000	1,450,000
0.60	45,810,000	0.97	0.69	1,430,000	1,010,000
0.70	33,280,000	1.10	0.69	1,170,000	740,000
0.80	24,750,000	1.22	0.70	970,000	560,000
0.90	18,600,000	1.34	0.71	800,000	420,000
1.00	14,210,000	1.46	0.71	670,000	320,000

Table 14-15 Spanish Mountain Inferred Resource

Au Cut-off (g/t)	Tonnes > Cut-off (tonnes)	Grade > Cut-off		Contained Metal	
		Au (g/t)	Ag (g/t)	Oz. Gold	Oz. Silver
0.10	691,530,000	0.23	0.59	5,070,000	13,140,000
0.15	**450,640,000**	**0.28**	**0.61**	**4,110,000**	**8,900,000**
0.20	307,410,000	0.34	0.63	3,320,000	6,250,000
0.25	203,740,000	0.39	0.65	2,580,000	4,240,000
0.30	136,250,000	0.45	0.66	1,980,000	2,900,000
0.40	61,590,000	0.59	0.69	1,160,000	1,360,000
0.50	32,180,000	0.72	0.69	740,000	710,000
0.60	18,410,000	0.85	0.67	500,000	400,000
0.70	11,280,000	0.99	0.68	360,000	250,000
0.80	7,590,000	1.10	0.68	270,000	170,000
0.90	4,920,000	1.24	0.68	200,000	110,000
1.00	3,400,000	1.37	0.67	150,000	70,000

Notes for Resource Tables:

- *Tonnages and Contained metals may not exactly equal individual tables due to rounding.*
- *This Mineral Resource Estimate was prepared by Gary Giroux, P.Eng. in accordance with CIM Definition Standards and NI 43-101, with an effective date of October 3, 2016.*
- *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Resource will ever be upgraded to a higher category.*
- *The estimate of Mineral Resources may be materially affected by environmental, permitting, legal or other relevant issues. The Mineral Resources have been classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves in effect as of the date of this Technical Report.*

14.8 Pit Delineated Resources

Pit delineated resources, which are based on the mineral resource, are described in Section 16.0. Pit delineated resources utilize a 0.15g/t gold cut-off grade. Subsequent mine planning and project economics treat the Inferred Class resource as waste rock.

Table 14-16 Pit Delineated Resources

Class	Total	Units
Measured Resource	39,650	kt
Gold Grade	0.54	g/t
Silver Grade	0.63	g/t
Indicated Resource	138,315	kt
Gold Grade	0.40	g/t
Silver Grade	0.68	g/t
Inferred Resource	21,226	kt
Gold Grade	0.30	g/t
Silver Grade	0.67	g/t
Waste Rock	235,888	kt
Total Pit Contents	435,079	kt

15.0 Mineral Reserve Estimates

As a PEA, there is no mineral reserve estimate.

16.0 Mining Method

The Spanish Mountain deposit is 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, WRSF (waste rock storage facility) 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 fourteen years of operation. The potential in-pit resource, summarized in Table 16-1 on a 0.15 g/t gold cut-off, forms the basis of the mine plan and production schedule. There is no certainty that the economic results from this PEA will be realized.

Table 16-1 Potential In-Pit Resource Estimate

	Unit	Amount
Measured and Indicated Resource Class	kt	177,968
Gold Grade	g/t	0.44
Silver Grade	g/t	0.67
Waste Material	kt	257,102
Strip Ratio	t/t	1.4

Note: Mineral resources that are not mineral reserves do not have demonstrated economic viability

The crusher will be fed with material from the pit and supplemented by the ROM stockpile in some years, at an average rate of 20,000t/d. After the pit is completely mined out, the ROM stockpile will feed the crusher for an additional thirteen years.

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 six 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 cut-off grade will be employed in the initial production years to enhance the economics of the project. Mineralized material that is below the elevated cut-off grade, but above the mine cut-off grade, will be sent to a stockpile near the crusher and either reclaimed at the end of the mine life (in Year 15), or blended with the run-of-mine (ROM) feed if an appropriate opportunity arises. Mineralized material that is below the mine cut-off grade, but of sufficient grade to cover the cost of milling and handling once it is hauled out of the pit, will also be sent to the mill, either directly or through the ROM stockpile.

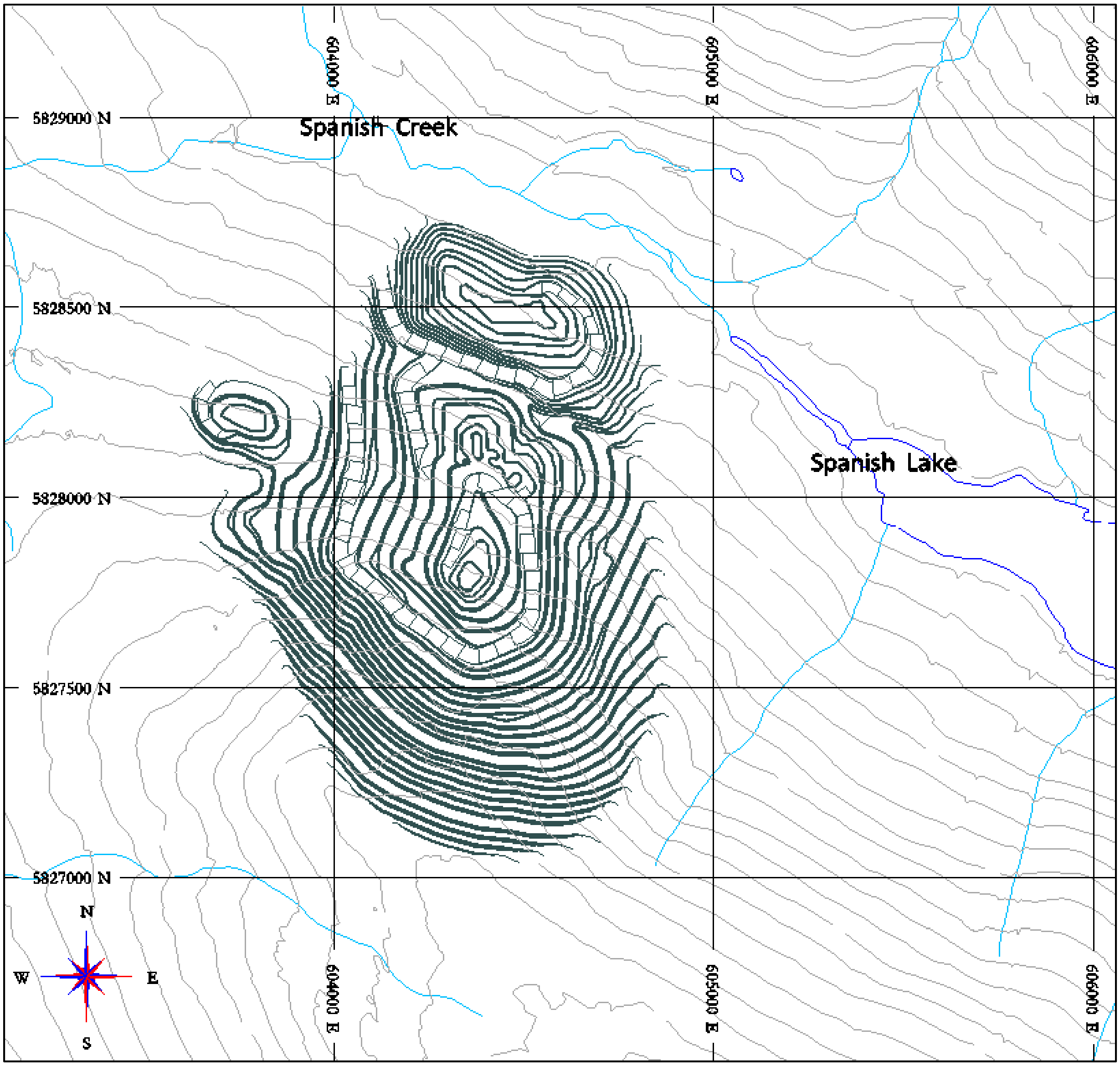


Figure 16-1 Ultimate Pit Design - Plan View

Most the pit waste material will be hauled to WRSF’s located on the west side of the pit. Some backfilling will also be available when the north end of the pit is completed later in the mine life. 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 TMF 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 3DBM provided by Giroux Consultants Ltd. as described in Section **Error! Reference source not found.**. Extra items are added to the original 3DBM to carry out open pit mine planning. Details of the mine planning 3DBM, including item descriptions, are included in the Appendix 16-1: Block Model Coding.

16.2.1 Mining Loss and Dilution

The mineralized material is represented in the resource 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. A value of 1% mining loss and dilution was 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 in the open pit resource estimate and 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. The NSP calculation is included in Appendix 16-2: NSP Calculations.

Using a gold market price of US$1,100/oz results in a NSP value of C$1,391/oz or C$44.73/g. Silver grades and associated net smelter value are not included in the LG analysis. The NSP calculation uses the inputs shown in Table 16-2:

Table 16-2 NSP Calculation Inputs

Description	Values	Units
Gold Price	$1,100	US$/oz
US Exchange rate	0.77	US$/C$
Payable Au	99.5%	%
Au Refining	$8.00	US$/oz
Royalty	1.5%	

16.3.1 Process Recovery

The process recovery assumptions are 90% for both the pit optimization and cut-off grade estimation.

16.3.2 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 16-3 LG Operating Cost Inputs

Operation	Cost
Base Ore Mining Cost (Pit Rim)	\$1.70/t
Base Waste Mining Cost (Pit Rim)	\$1.80/t
Incremental Haulage Cost	\$0.015/5 m bench below 950 m model elev.
Processing Cost	\$4.90/t
General/Administration Cost	\$1.17/t

16.3.3 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 16-4 and Figure 16-2 summarizes the wall design criteria for each sector. BGC's report in its entirety is provided in Appendix 16-5. In-pit ramps (37 m wide) and geotechnical berms (minimum 20 m wide) are included in the design where necessary to reduce the overall slope angles and facilitate geotechnical instrumentation and dewatering.

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 Table 16-4.

Table 16-4 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

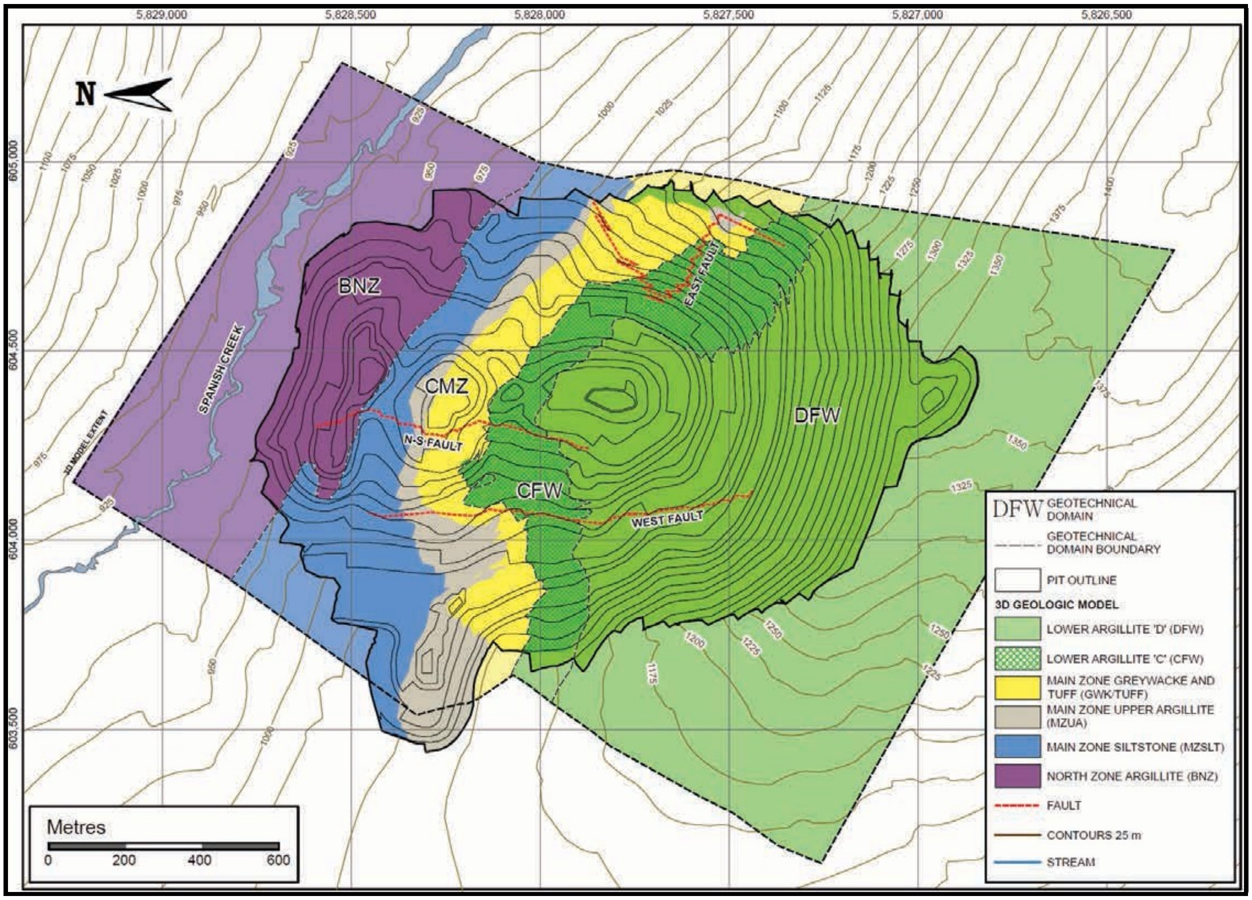


Figure 16-2 Structural Domains for Pit Slope Designs

16.3.4 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 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 gold cut-off grade is 0.15 g/t.

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 resource contents of the generated LG Price Case pit shells. An inflection point can be seen in the curve drawn in Figure 16-3 of cumulative resources by pit case. This inflection point is at the pit shell generated at the 86% case, and it indicates a point at which larger pit shells will not produce added economic returns to the project.

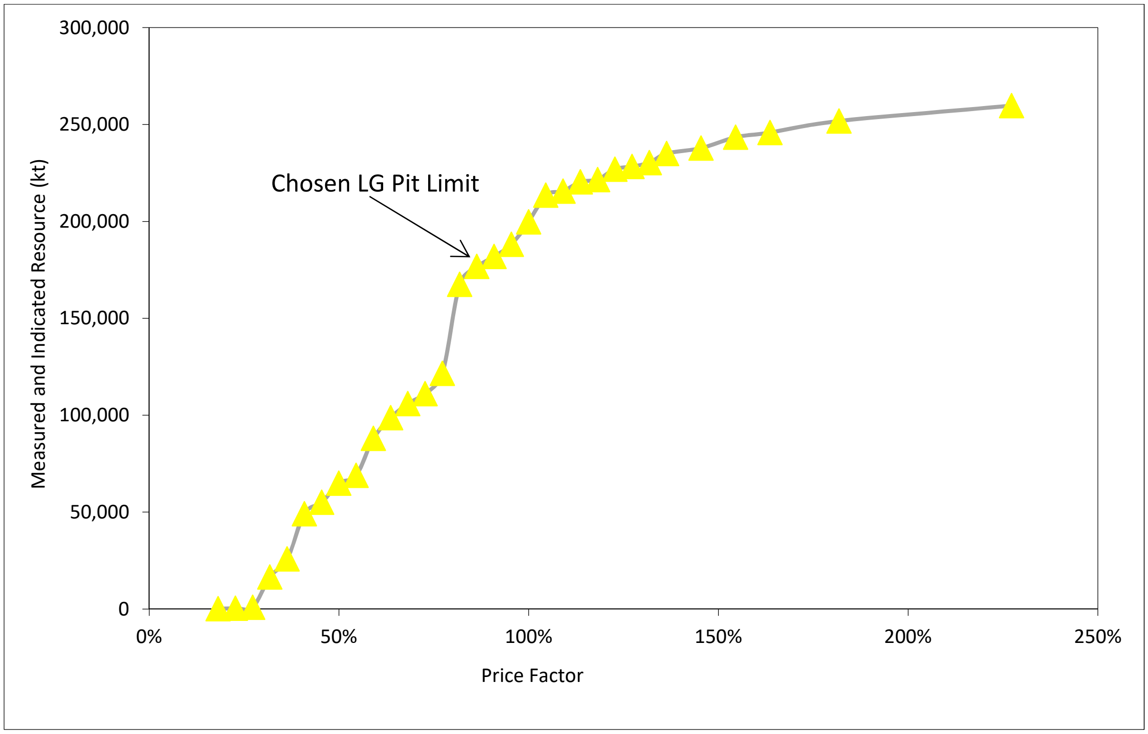


Figure 16-3 **LG Price Cases Cumulative Resources**

16.5.1 Selected Ultimate Pit Limits

The pit shell generated at the 86% case is selected as the ultimate pit limit, and is used for subsequent mine planning, in this study. The LG pit limit resource for Spanish Mountain Gold is shown in the Table below.

Table 16-5 **LG Pit Delineated Contents**

Input Gold Price	$950	US$/oz
Measured and Indicated Resource	176,769	kt
Gold Grade	0.44	g/t
Waste and Inferred Resource	266,626	kt
Strip Ratio	1.51	Waste / Resource
Total Pit Contents	443,395	kt

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

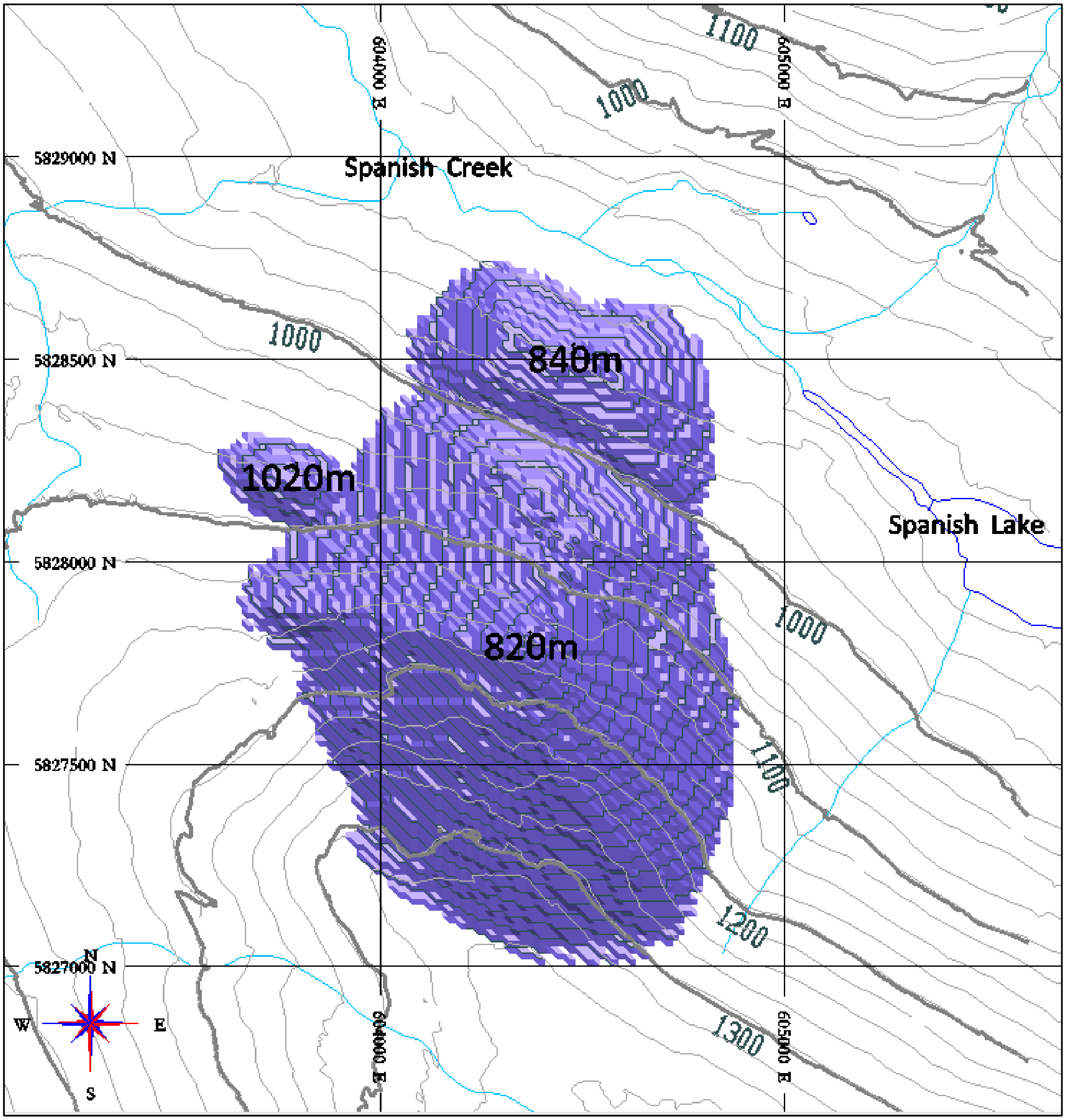


Figure 16-4 Plan View of Optimized Pit LG Shell

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.

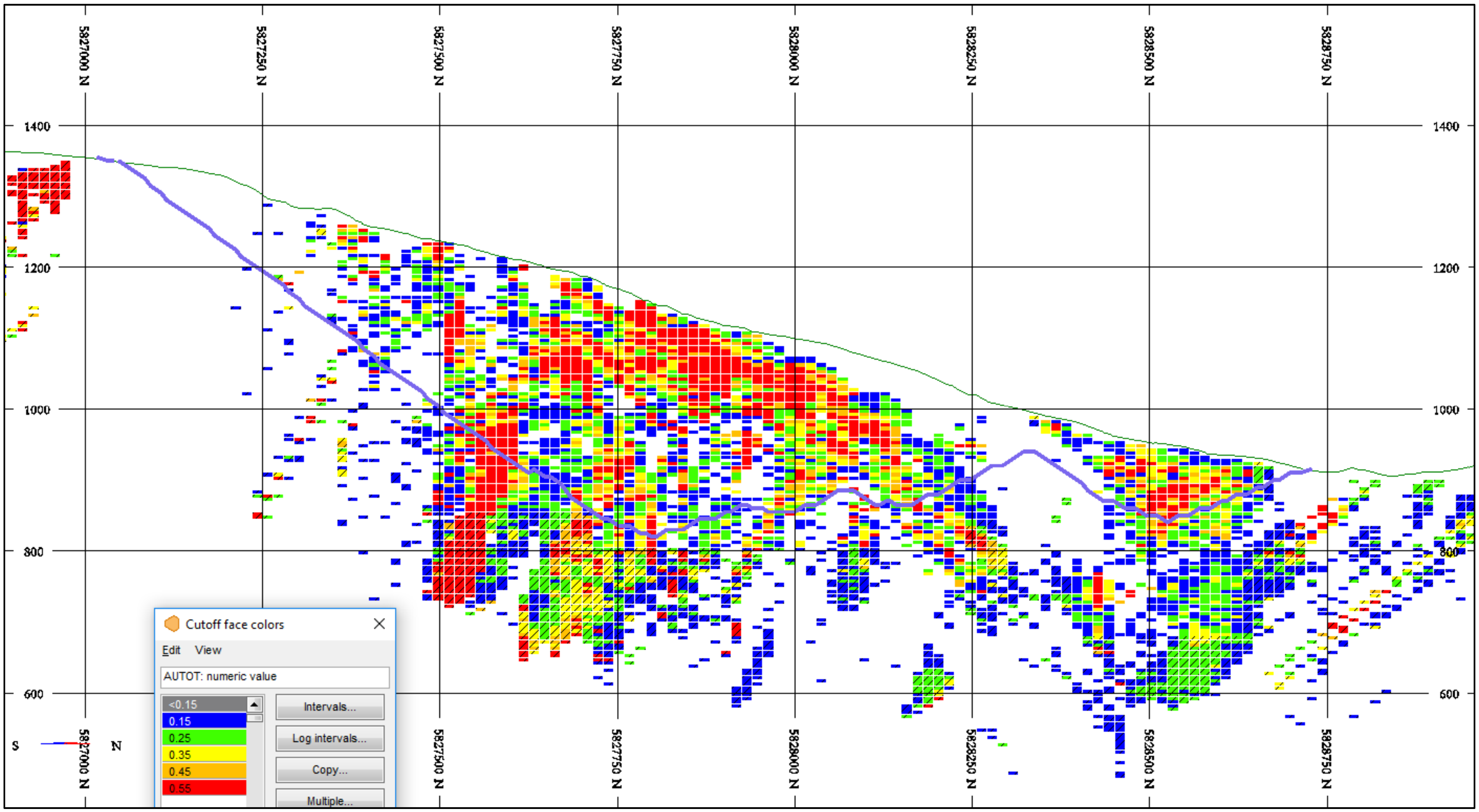


Figure 16-5 **Cross Section View, 604370E (looking west), of Optimized Pit LG shell**

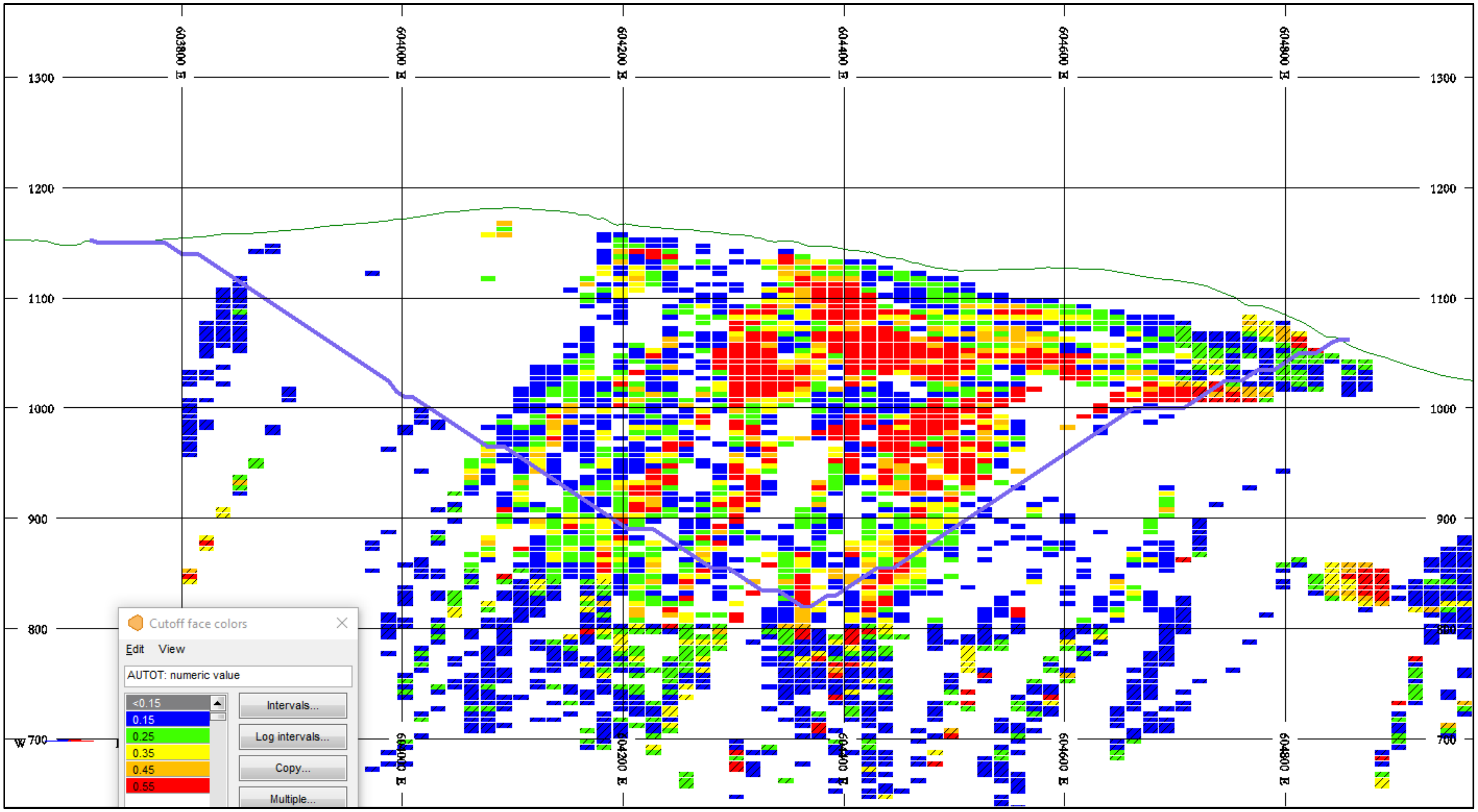


Figure 16-6 Cross Section View, 5,827,805N (looking north), of Optimized Pit LG shell

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 sufficient resource to sustain the plant operations for at least two years (15 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 100 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 36% gold price factor is used to target a starter pit phase. The remaining phases are designed to mine 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 16-4 and Figure 16-2.
- Suitable single and dual lane haul road widths to handle 180 t payload haulers.
  - 23 m for single lane traffic
  - 35 m for dual lane traffic
- The ramp is not extended into the bottom 20 m of the pit, assuming the ramp will be retreat mined out of these benches.
- The ramp is designed to single lane width for the 20 m above this, assuming single lane traffic is adequate.
- 10% maximum ramp grade.
- Pit exits face west towards the crusher, WRSF's 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 pit phase number.
- The final digit signifies the design series.

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, P611

This designed pit contains just over three 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 1255 m elevation, down to the pit exit at the 1055 m elevation via external roads; then down the ramp to the pit bottom at the 995 m elevation. The upper benches will be mined in conjunction Phase 2 to the 1025 m elevation, and Phase 5 to the 1175 m elevation.

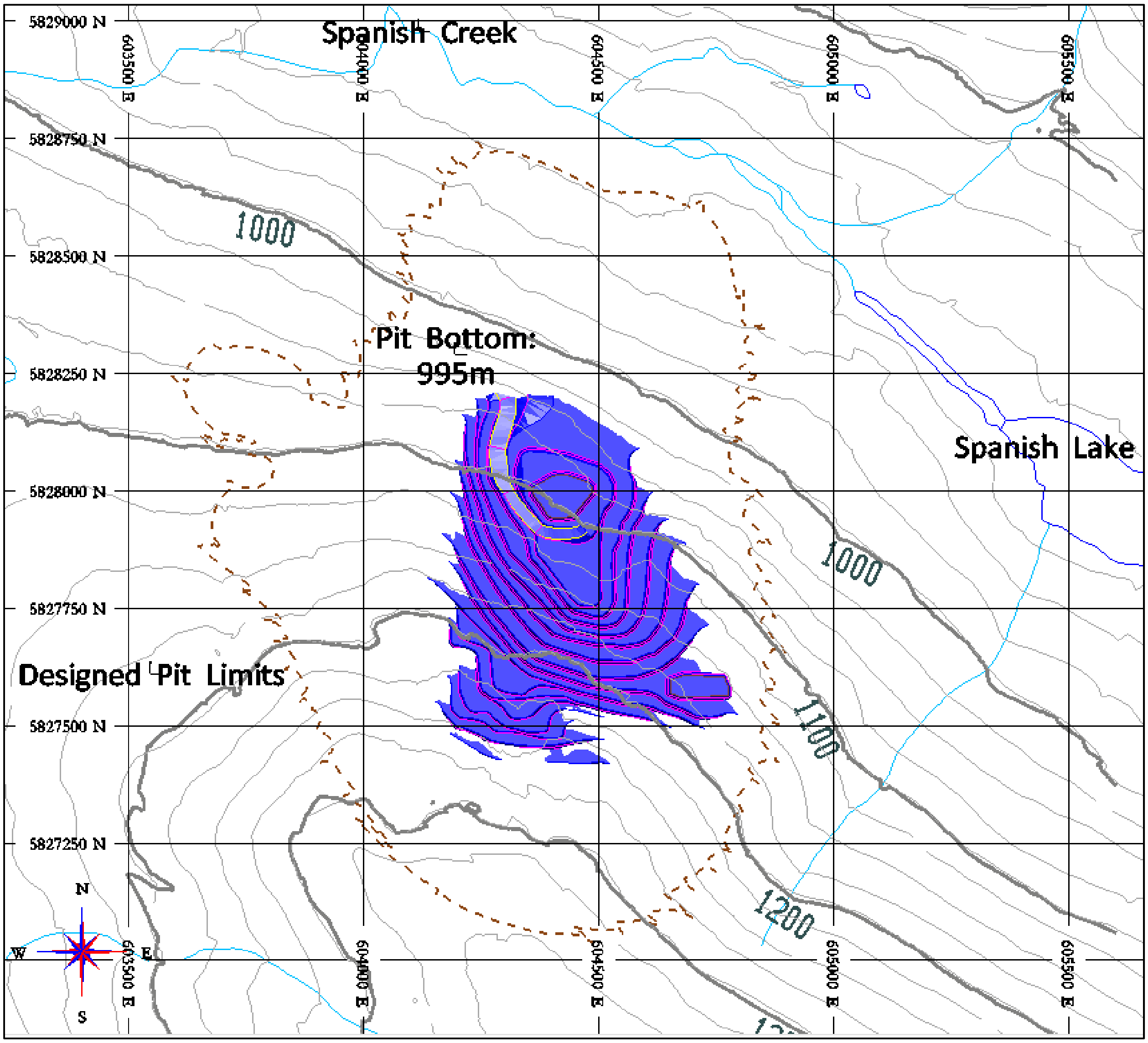


Figure 16-7 Plan View of Phase 1, P611

16.7.2 **Phase 2, P621**

Phase 2, P621 is a north and east pushback on the P611 pit. This pit mines from the pushback crest at 1150 m elevation down to the pit exit at the 1025 m elevation via external roads; then down the ramp to the pit bottom at the 905 m elevation. This pit phase will be mined concurrently with Phase 1 down to the pit exit at the 1025 m elevation.

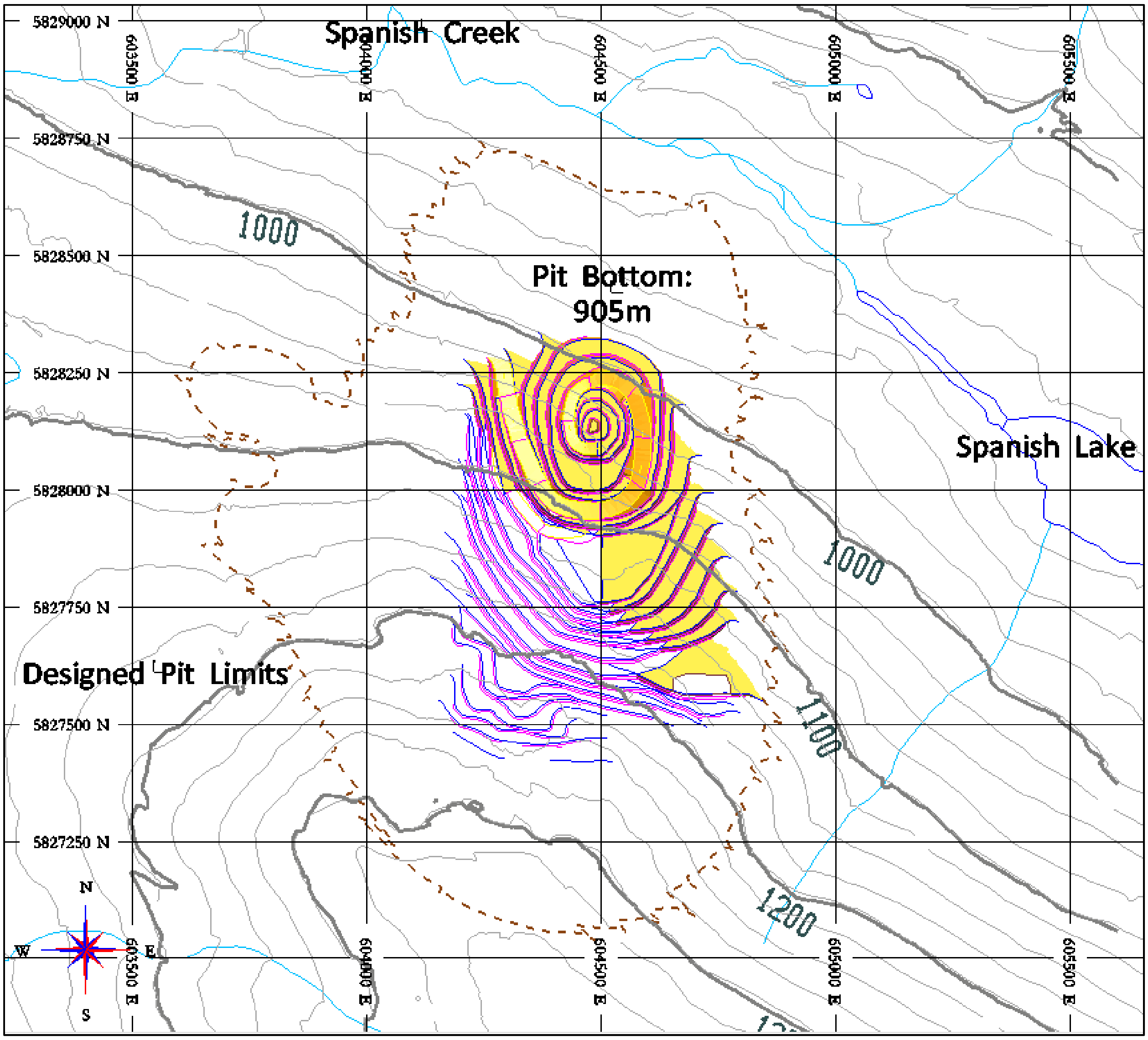


Figure 16-8 **Plan View of Phase 2, P621**

16.7.3 Phase 3, P631

Phase 3, P631 encompasses northern portion of the designed pit limits. 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 965 m elevation via external roads; then down the ramp to the pit bottom at the 845 m elevation. This pit phase will be mined concurrently with Phase 1 down to the pit exit at the 1025 m elevation. This phase is mined independently of all other phases. Once mined out, this area can be used for waste rock storage.

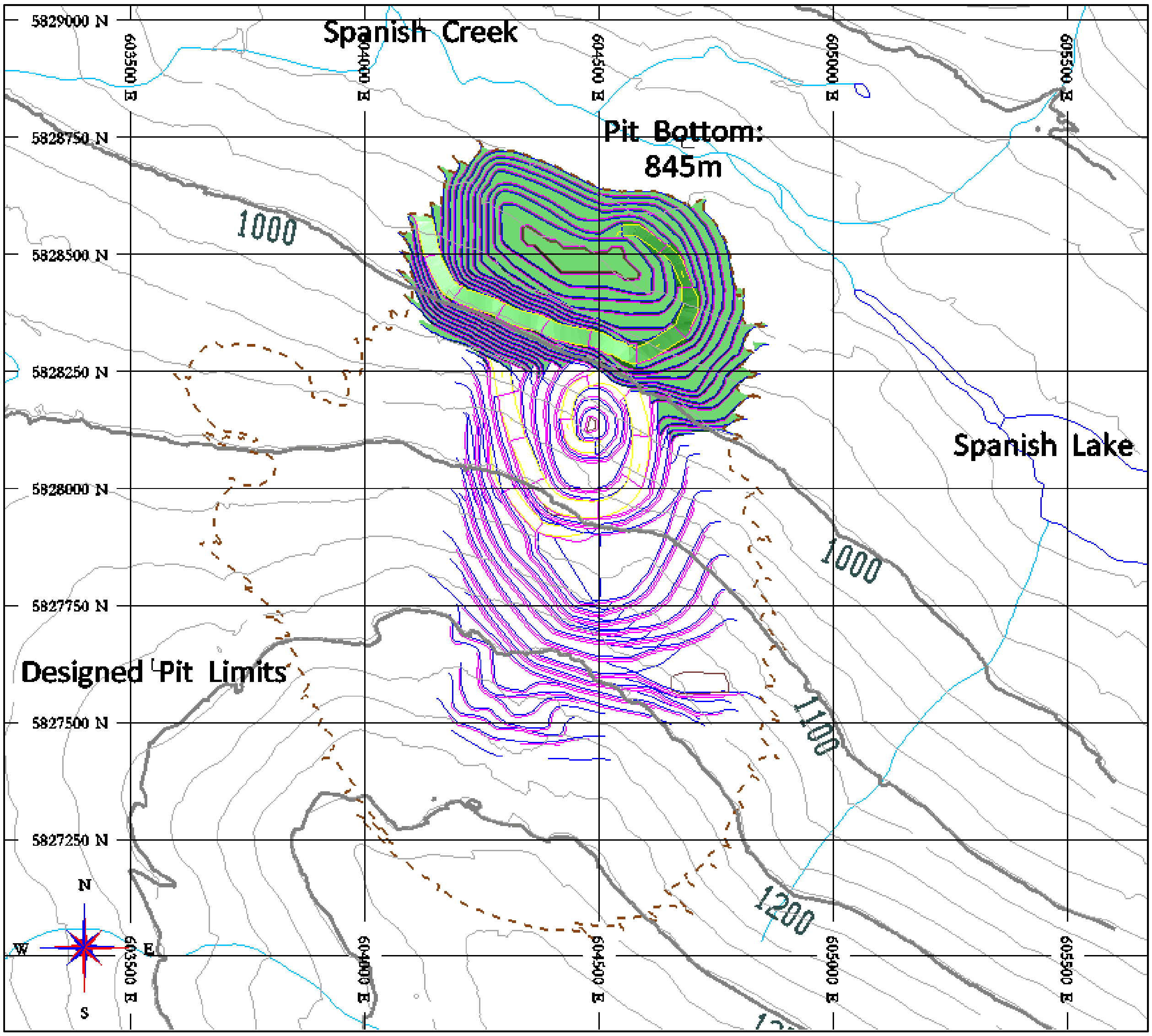


Figure 16-9 Plan View of Phase 3, P631

16.7.4 Phase 4, P641

Phase 4, P641 is a mini pit at the western edge of the designed pit limits. This pit mines from the crest at 1105 m elevation down to the pit exit at the 1060 m elevation via external roads; then down the ramp to the pit bottom at the 1025 m elevation. This phase is mined independently of all other phases.

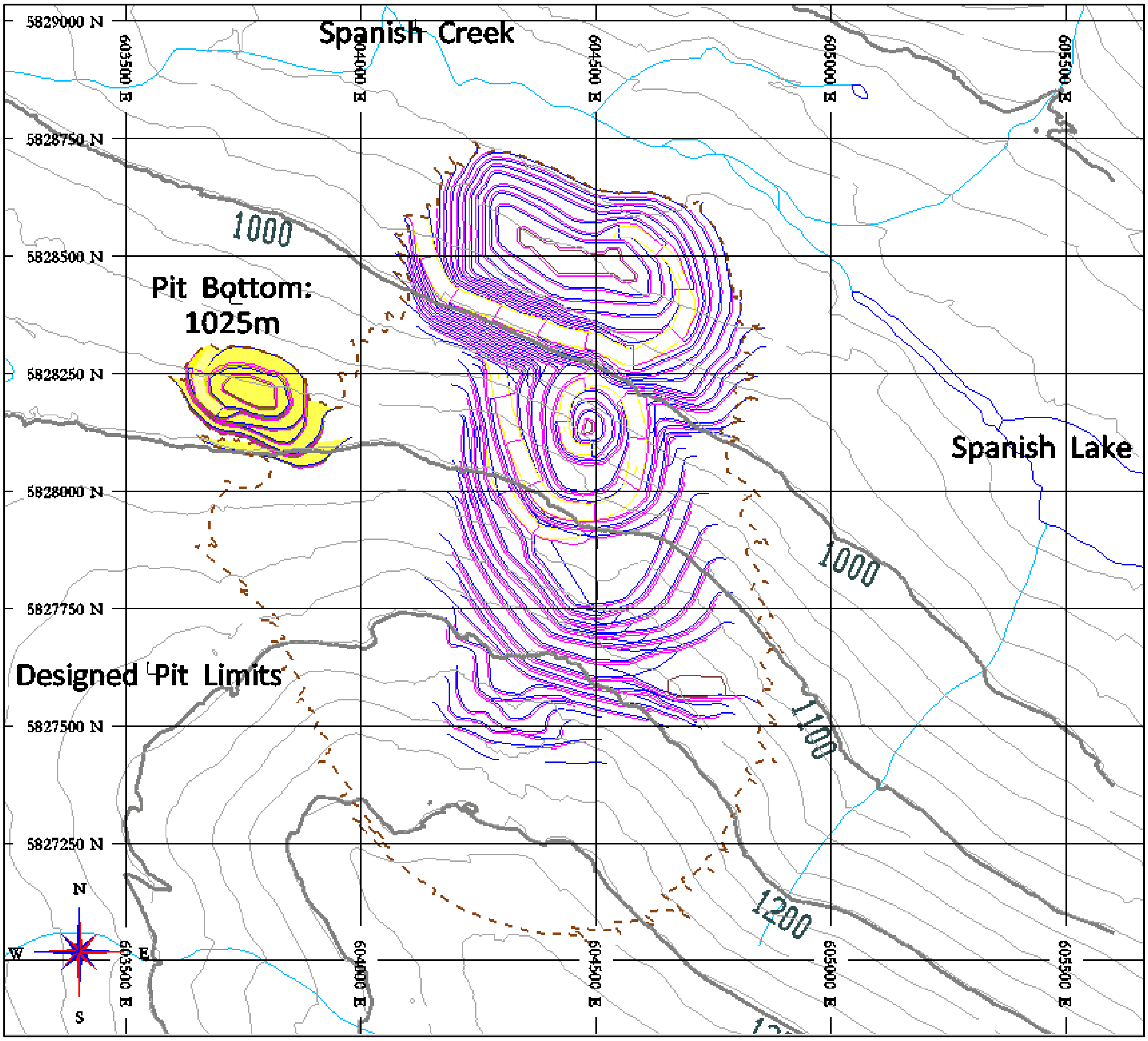


Figure 16-10 **Plan View of Phase 4, P641**

16.7.5 Phase 5, P651

Phase 5, P651 is a west, south and east pushback off the phase 2 pit. The south highwall is kept just inside the "CFW" geotechnical zone. This pit mines from the crest at 1275 m elevation down to the pit exit at the 965 m elevation via external roads; then down the ramp to the pit bottom at the 865 m elevation. From the 965 m elevation to the 940 m elevation the ramp developed for phase 3 is utilized. The upper benches will be mined in conjunction Phase 1 to the 1175 m elevation.

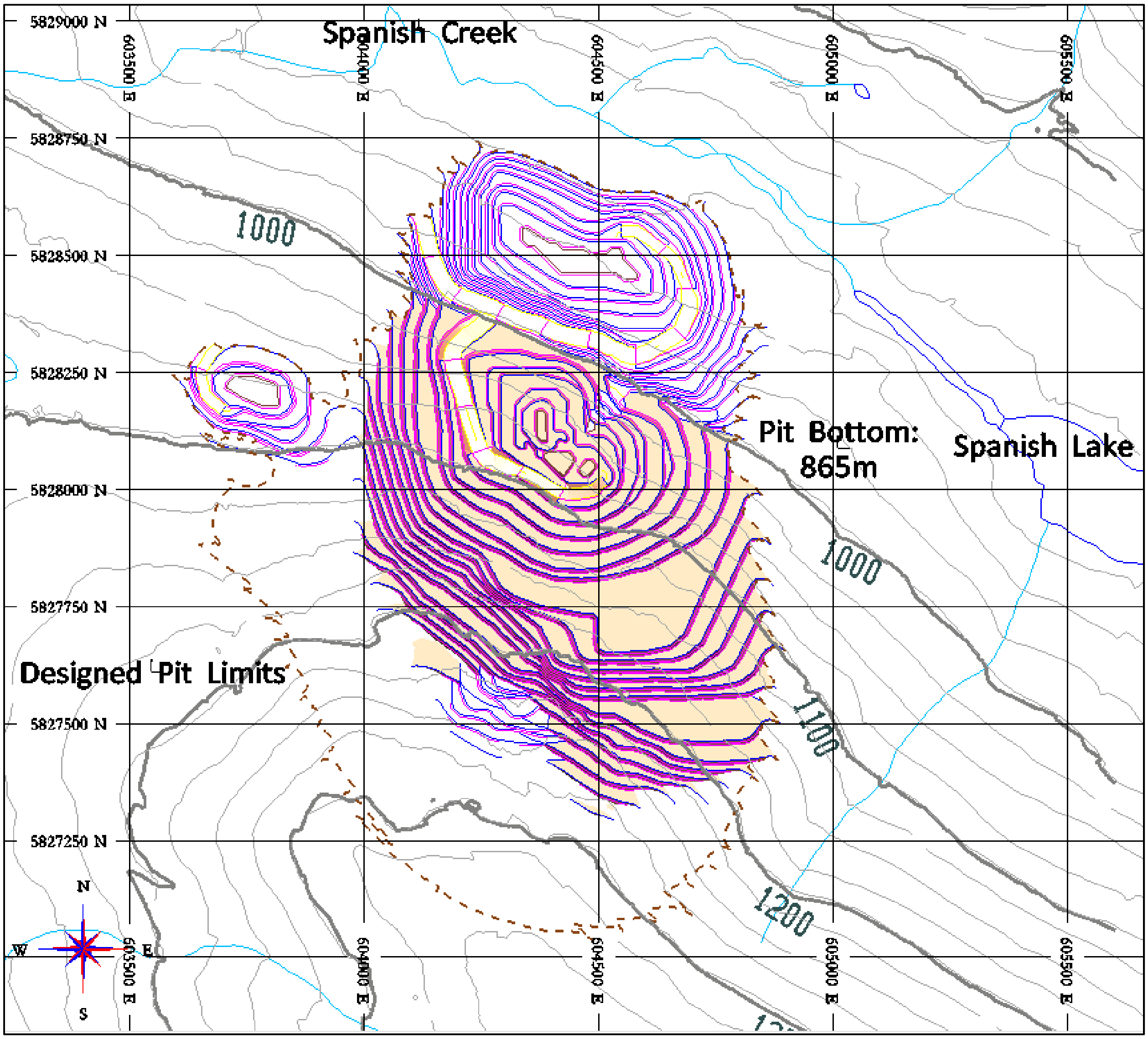


Figure 16-11 Plan View of Phase 5, P651

16.7.6 Phase 6, P661

Phase 6, P661 is a west and south pushback off the phase 5 pit to the designed pit limits. This pit mines from the crest at 1345 m elevation down to the pit exit at the 1025 m elevation via external roads; then down the ramp to the pit bottom at the 825 m elevation.

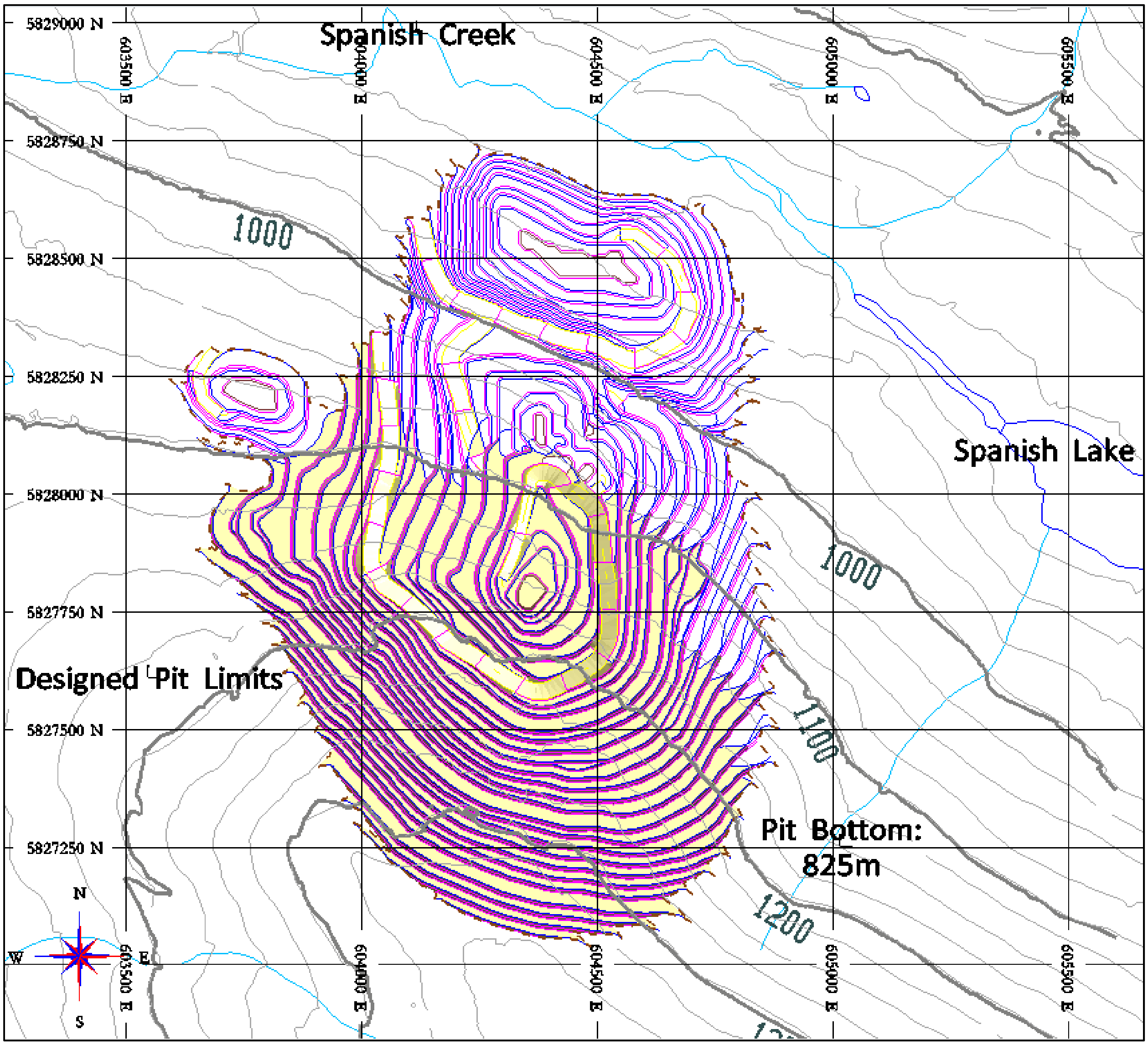


Figure 16-12 Plan View of Phase 6, P661

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.

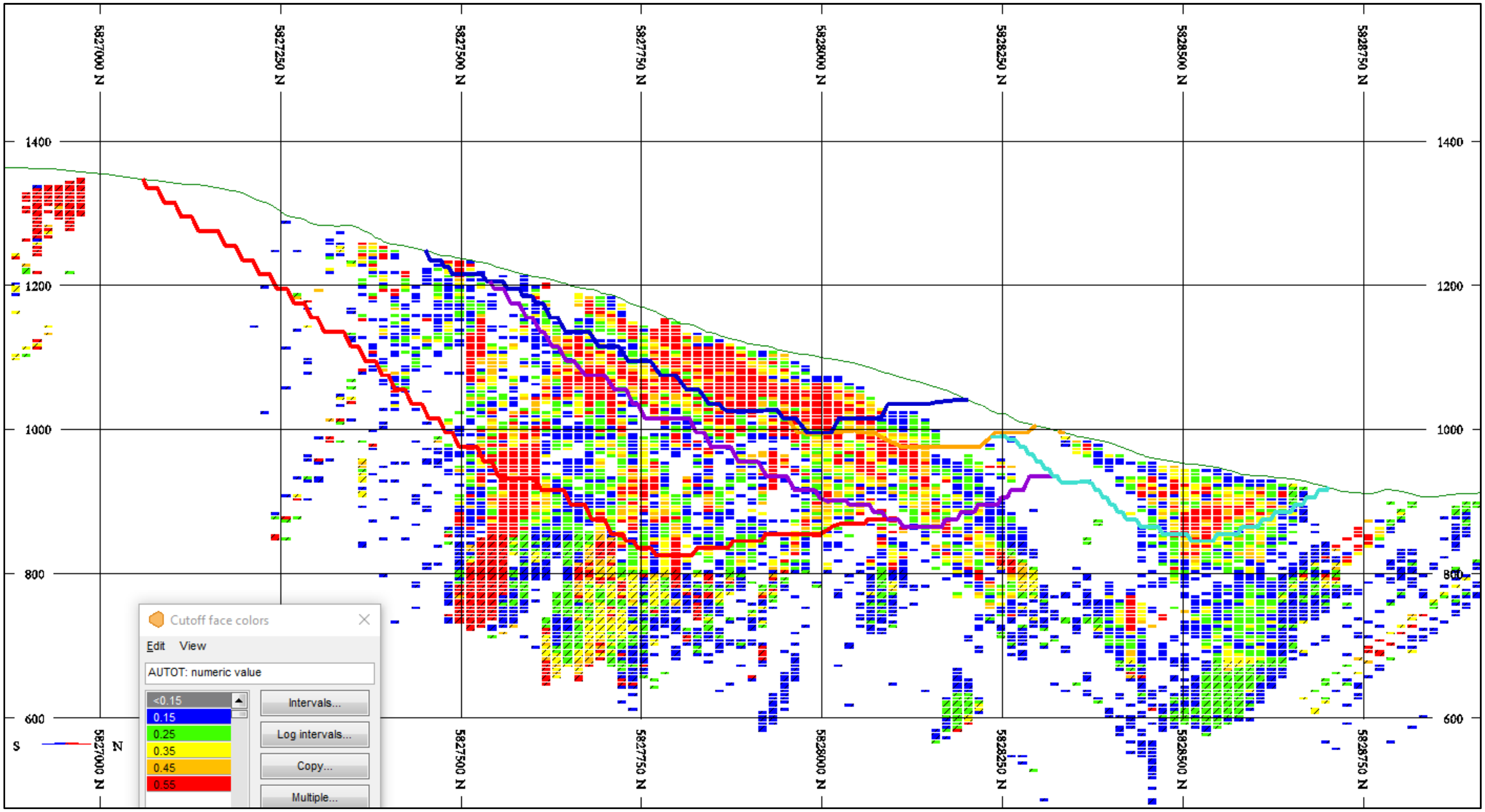


Figure 16-13 Cross Section View, 604370E (looking west) of Phased Pit Designs (P611 in blue, P621 in orange, P631 in cyan, P651 in purple, and P661 in red)

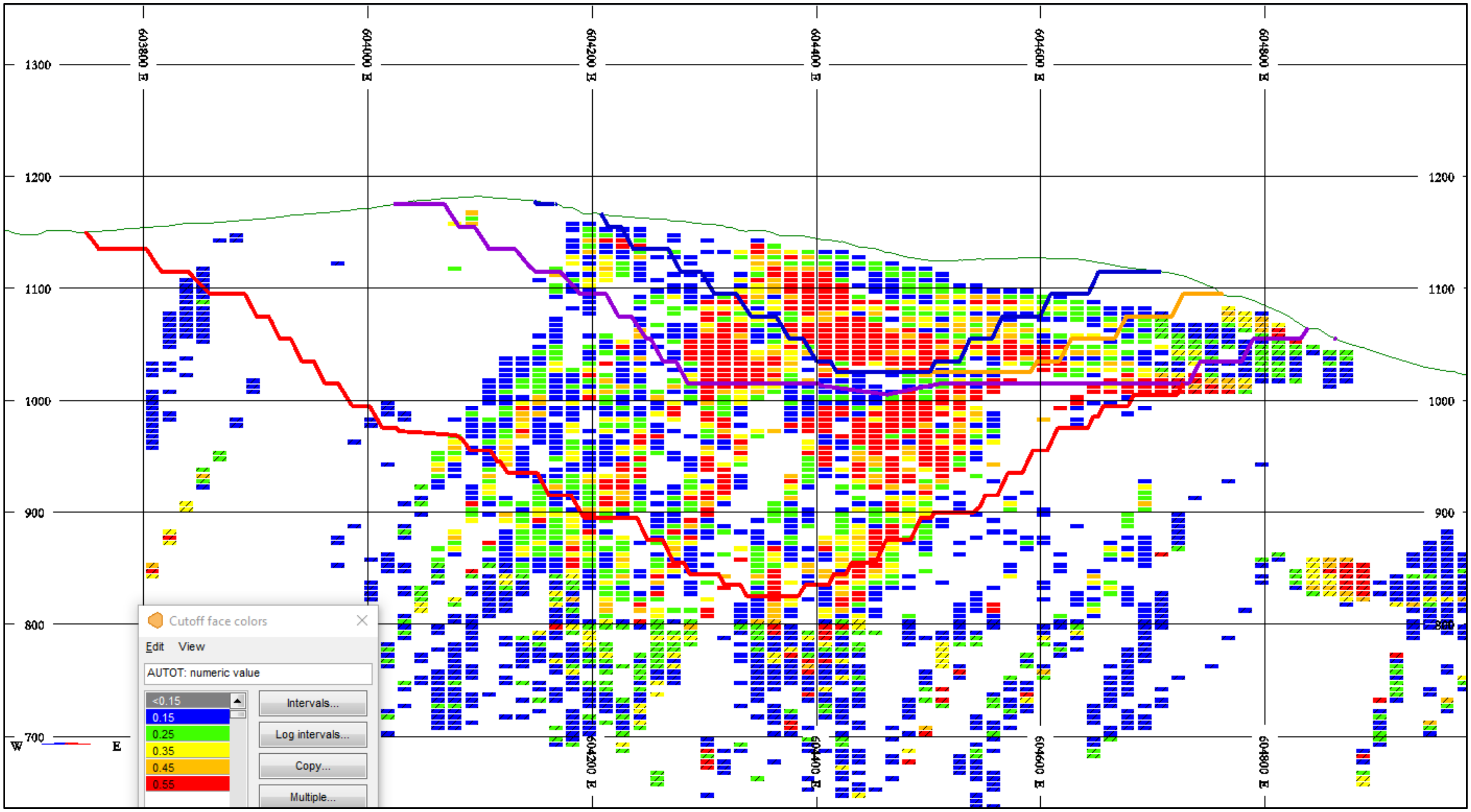


Figure 16-14 **Cross Section View, 5,827,805N (looking north), of Phased Pit Designs (P611 in blue, P621 in orange, P631 in cyan, P651 in purple, and P661 in red)**

16.8 Pit Resources

The resources delineated by the pit designs are shown in Table 16-6 . The utilized cut-off gold grade is 0.15g/t, and Inferred resources are treated as waste rock.

Table 16-6 Phased and Total Pit Delineated Resources

Pit Name	Units	P611	P621i	P631i	P641i	P651i	P661i	Total
Measured Resource	kt	10,888	1,356	3,074	0	15,727	8,605	**39,650**
Gold Grade	g/t	0.64	0.53	0.45	0.00	0.53	0.48	**0.54**
Indicated Resource	kt	14,127	9,075	16,084	1,081	34,629	63,319	**138,315**
Gold Grade	g/t	0.53	0.45	0.37	0.41	0.38	0.39	**0.40**
Wasted Inferred Resource	kt	40	169	2,916	358	2,793	14,950	**21,226**
Gold Grade	g/t	0.39	0.25	0.29	0.32	0.31	0.30	**0.30**
Waste	kt	12,682	13,393	18,629	1,853	48,796	140,528	**153,921**
Mill Feed (M+I Resource)	kt	25,015	10,431	19,158	1,081	50,356	71,924	**177,970**
Gold Grade	g/t	0.58	0.46	0.38	0.41	0.43	0.40	**0.44**
Silver Grade	g/t	0.78	0.88	0.78	0.48	0.60	0.62	**0.67**
Waste	kt	12,722	13,562	21,545	2,211	51,589	155,478	**257,109**
Strip Ratio	Wst. / Res.	0.51	1.30	1.12	2.05	1.02	2.16	**1.44**
Total Pit Contents	kt	37,737	23,993	40,703	3,292	101,945	227,402	**435,079**

Note: Mineral resources that are not mineral reserves do not have demonstrated economic viability

Tables of resource by bench are included in Appendix 16-3: Bench Resources by Phase

16.9 Waste Rock Management

16.9.1 Waste Rock Characterization

Sulphur, calcium, and arsenic values were interpolated into the 3DBM to quantify the acid rock drainage (ARD) generating potential of the pit material. SRK Consulting (Canada) Inc. (SRK) provided the following formulas and criteria to categorize the pit waste rock in 2012. SRK's memo to SMG regarding the ML/ARD block model criteria is provided in Appendix 16-6.

The ARE 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 (76%) and Bi (10%), 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.

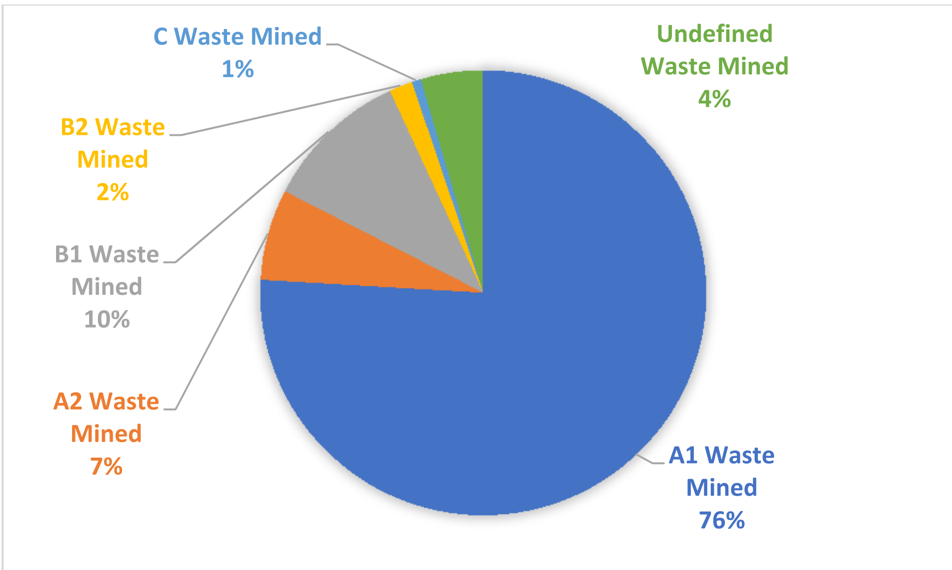


Figure 16-15 Pit Waste Rock Distribution by ARD Characterization

16.9.2 Waste Rock Suitability

The possible waste rock destinations are determined by their ARD generating characterization. There are three 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. Near the end of the mine life, there will be available space for an in-pit WRSF at the north end.

A placed waste density of 2.33 t/m^3 is assumed for all waste rock, and 1.77 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 2:1 slope (26.5 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 76 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 1,060 m, containing 37 Mt of waste rock.

Development at the north end of the pit will be completed by Year 4 and will be available for backfilling. Most waste rock after Year 8 will be placed at this location. The top elevation will be 1,015 m, containing 45 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 tailings facility for dam embankment construction as required. It is estimated that 35 Mt of material (30 Mt rock, 5 Mt overburden) will be required from the pit for both the north and south dam embankments through the LOM, including 7 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 43 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.

All Inferred class resources have been treated as waste rock, but for the purposes of this study, have been disposed of in a specific dump position south of the crusher. The top elevation of this pile is 1,120 m, containing 21 Mt of waste rock.

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%, and 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 three separate stockpiles will be maintained to store these resources for later re-handle to the crusher.

A "High-Grade (HG)" stockpile is built to the south of the crusher. This stockpile is used to feed the crusher during open pit operation, specifically at mill start-up and in Years 7 to 9 as the final pit phase is being pre-stripped. This will be the first stockpile to be fully reclaimed to the crusher once the open pit operations are completed.

A "Mid-Grade (MG)" stockpile is built to the southwest of the crusher, next to the HG stockpile. This stockpile will not be reclaimed from until open pit operations are completed, and the High-Grade stockpile is completely mined out.

A "Low-Grade (LG)" stockpile is built north of the plant site, and west of the north WRSF. This stockpile will not be reclaimed from until open pit operations are completed, and both the High-Grade and Mid-Grade stockpiles are completely mined out. When reclamation from this stockpile is commencing, it is anticipated that the crusher will be moved nearby to facilitate short haul cycles.

Table 16-7 Stockpile Bins and Capacities

Stockpile	Gold Grade Bin (g/t)	LOM to SP (t)	Maximum Size (t)
High Grade	>0.35	11,658	8,950
Mid Grade	0.25 - 0.35	36,201	36,201
Low Grade	0.15 - 0.25	59,720	59,720

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 is 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.

Calculations for drill, blast, load and haul productivities are shown in Appendix 16-4: Mine Productivities.

16.12.2 **Drilling**

Diesel powered rotary drills capable of drilling 255 mm diameter holes will be used for production drilling. Two 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 20 to 25 m^3 sized hydraulic shovels, with the capability to excavate in both front and backhoe configurations, will meet these requirements. One wheel loader with an 18 m^3 bucket is also specified for pre-production 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 is 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 180-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.

16.12.6 **Pit Support Services**

Pit Support Services include:

- Haul road development and maintenance
- Pit floor and ramp maintenance
- WRSF maintenance
- Ditching
- Reclamation
- 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, backhoes, and wheel loader and a scraper for pit floor maintenance, road development and maintenance, and ditching. The road maintenance fleet will also include motor graders and a water/sanding truck.

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, Geotechnical and Environmental 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 first pre-production period to access the top benches of the pit from the crusher and stockpile, and from the pit to the tailings facility and WRSF's.

A pre-strip of 11.3 Mt of is required. The construction of the starter tailings dam will require 7.0 Mt during the pre-strip. The remainder is associated resource material that will be stockpiled, 3.0 Mt, and waste rock to build haul roads.

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.

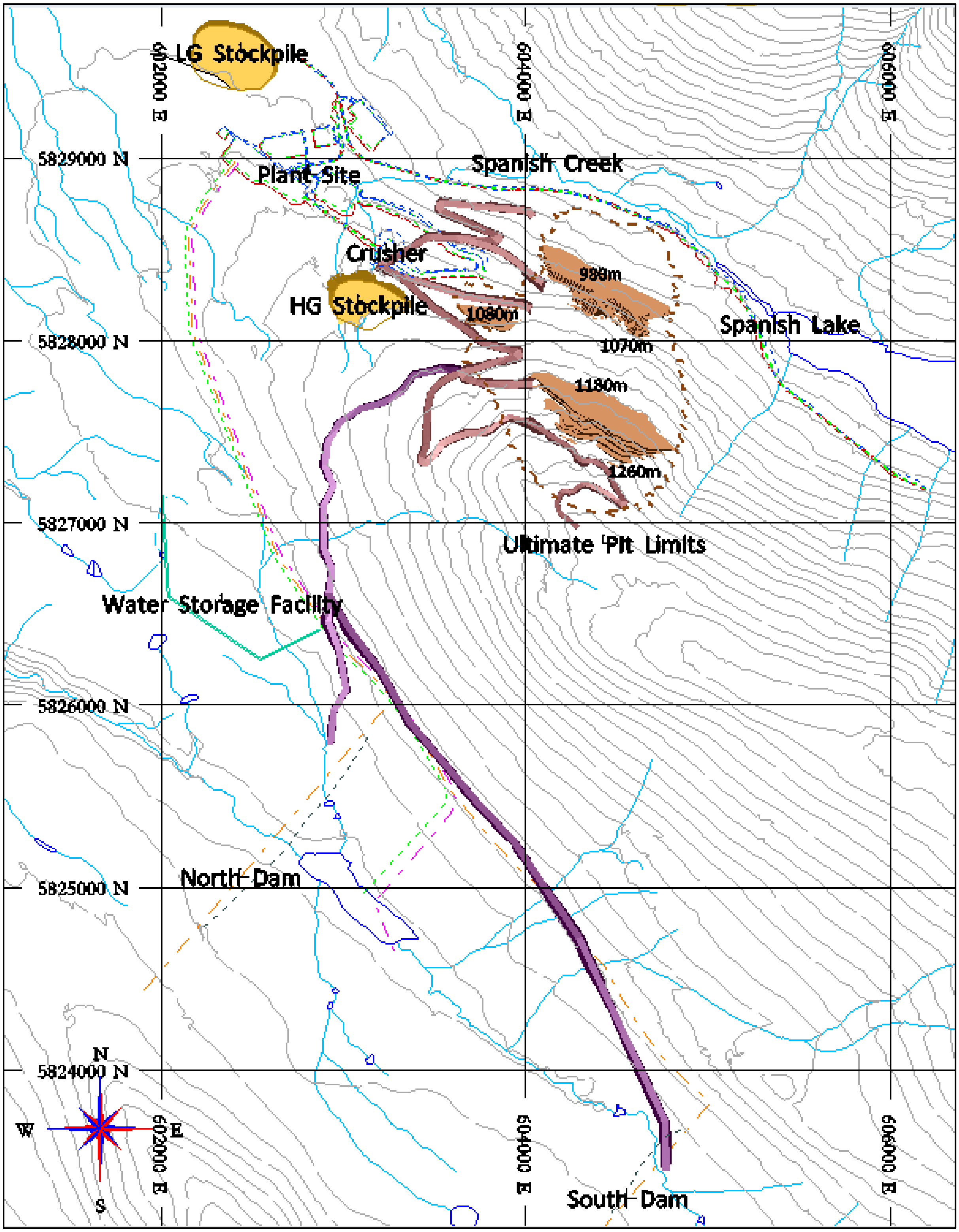


Figure 16-16 Mine Development - Pre-Production

16.13.2 **Production Scheduling**

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

- Annual mill feed of 7,300,000 t is targeted based on an average of 20,000 tonnes/day milling.
  - Mill production will ramp up through Year 1, totalling 6,000,000 t.
- Phased pit bench resources and waste rock contents are used as input to the mine production schedule.
- Within a given phase, each bench is fully mined before progressing to the next bench. Optimization by partial bench mining is not examined at this level of study, even in zones of predominately waste rock.
- Pit phases are mined sequentially, with more than one phase in production in a period. A following phase is limited from progressing vertically beyond its predecessor.
  - Exceptions are P631 and P641 which are progressed independently of the other phases.
- Pit phase progression is limited to no more than ten benches in a given year. Average phase progression in an annual period is 6 benches.
- A cut-off grade strategy is employed. An elevated gold cut-off grade is employed during the open pit operations.
  - Years 1 to 5 of 0.40 g/t
  - Years 6 to 12 of 0.35 g/t
- Material in the stockpiles is reclaimed to the mill after the pit is mined out in Y12.

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 Rock (kt)	Strip Ratio (t:t)	Total Mined (kt)	Total Moved (kt)
Y-2		832					2,461	3.0	3,292	3,292
Y-1		2,198					5,858	2.7	8,056	8,056
Y1	4,749	5,949	1,251	6,000	0.77	0.72	8,598	0.7	19,296	20,547
Y2	7,300	4,702		7,300	0.83	0.73	7,784	0.6	19,786	19,786
Y3	7,300	13,102		7,300	0.73	0.77	18,651	0.9	39,053	39,053
Y4	7,300	12,504		7,300	0.73	0.68	22,701	1.1	42,505	42,505
Y5	7,300	13,227		7,300	0.80	0.57	22,266	1.1	42,794	42,794
Y6	7,300	12,180		7,300	0.69	0.61	29,662	1.5	49,142	49,142
Y7	3,700	5,177	3,600	7,300	0.52	0.71	40,341	3.2	49,218	52,818
Y8	3,300	6,713	4,000	7,300	0.51	0.70	38,980	2.8	48,992	52,992
Y9	5,950	7,298	1,350	7,300	0.55	0.64	31,808	2.2	45,056	46,406
Y10	7,300	13,607		7,300	0.74	0.66	20,391	1.0	41,299	41,299
Y11	7,300	8,889		7,300	0.65	0.62	6,848	0.4	23,037	23,037
Y12	1,590	1,202	5,711	7,300	0.36	0.65	753	0.1	3,544	9,255
Y13			7,300	7,300	0.29	0.69				7,300
Y14			7,300	7,300	0.29	0.69				7,300
Y15			7,300	7,300	0.29	0.69				7,300
Y16			7,300	7,300	0.29	0.69				7,300
Y17			7,300	7,300	0.23	0.67				7,300
Y18			7,300	7,300	0.19	0.66				7,300
Y19			7,300	7,300	0.19	0.66				7,300
Y20			7,300	7,300	0.19	0.66				7,300
Y21			7,300	7,300	0.19	0.66				7,300
Y22			7,300	7,300	0.19	0.66				7,300
Y23			7,300	7,300	0.19	0.66				7,300
Y24			7,300	7,300	0.19	0.66				7,300
Y25			4,067	4,067	0.19	0.66				4,067
Total	**70,389**	**107,579**	**107,579**	**177,968**	**0.44**	**0.67**	**257,102**	**0.9**	**435,071**	**542,650**

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

Period	P611, Starter Pit (kt)	P621, North Pushback (kt)	P631, North Pit (kt)	P641, West Pit (kt)	P651, South Pushback (kt)	P661, Final Pushback (kt)	Total (kt)
Y-2	1,152			441	1,700		3,292
Y-1	2,222		2,179		3,654		8,056
Y1	17,419	1,877					19,296
Y2	12,978	4,067	2,741				19,786
Y3	3,964	2,839	32,250				39,053
Y4		13,240	3,534		25,731		42,505
Y5		1,209		2,851	38,733		42,794
Y6		764			31,740	16,637	49,142
Y7					382	48,836	49,218
Y8						48,992	48,992
Y9						45,056	45,056
Y10						41,299	41,299
Y11						23,037	23,037
Y12						3,544	3,544
Total	**37,735**	**23,996**	**40,704**	**3,293**	**101,940**	**227,403**	**435,071**

Table 16-10 Waste Rock Quantities by Destination

Period	Dam Embankment (kt)	Sub-Aqueous Disposal (kt)	Waste Dumps (kt)
Y-2	1,800	53	608
Y-1	5,150	140	568
Y1	3,300	4,248	1,050
Y2	3,500	3,400	884
Y3	3,150	4,215	11,286
Y4	2,900	6,086	13,715
Y5	2,900	3,701	15,665
Y6	2,100	4,309	23,253
Y7	2,100	7,549	30,692
Y8	2,100	5,105	31,775
Y9	2,100	3,268	26,440
Y10	2,100	704	17,587
Y11	1,600	315	4,933
Y12		129	624
Total	**34,800**	**43,222**	**179,081**

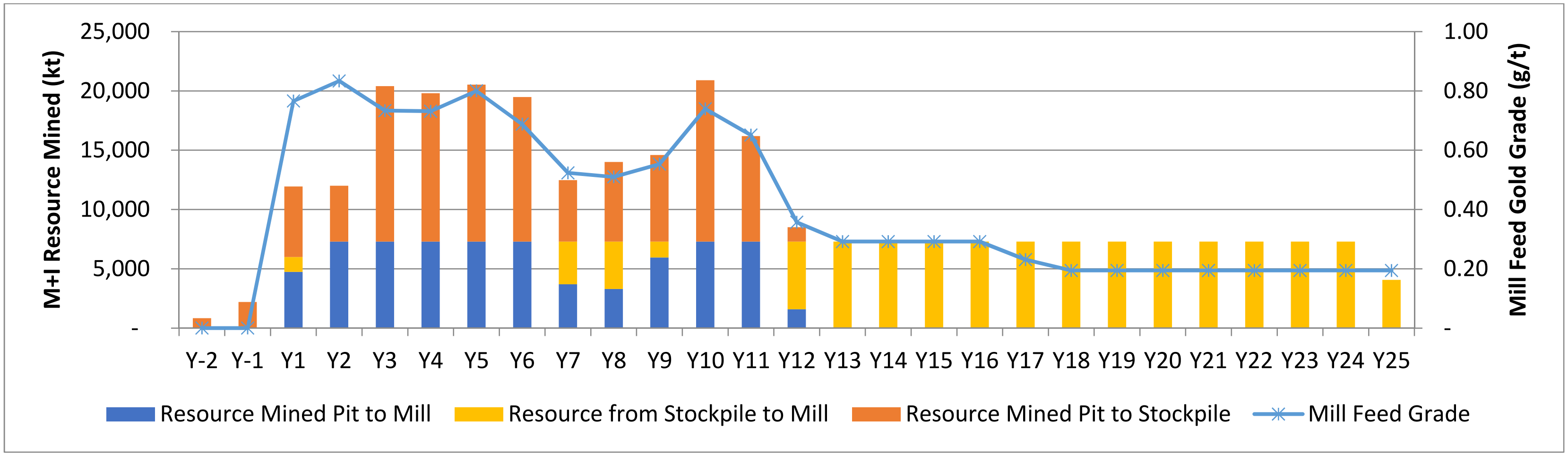


Figure 16-17 Mine Production Schedule, Resource Mined and Mill Feed Grades

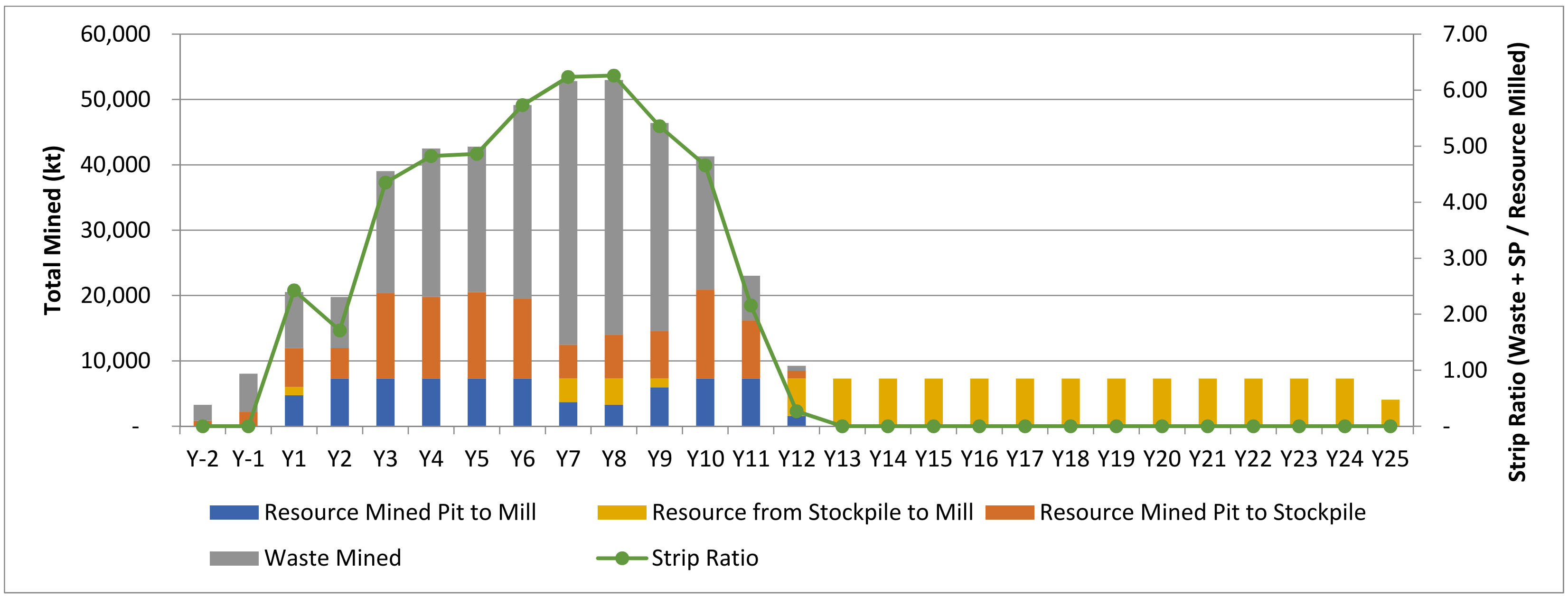


Figure 16-18 Mine Production Schedule, Total Mined and Strip Ratio

16.14 Mine End of Period Maps

The following figures show the general arrangement of the mine operations at Year 2, Year 5 and Year 12, the completion of open pit operations.

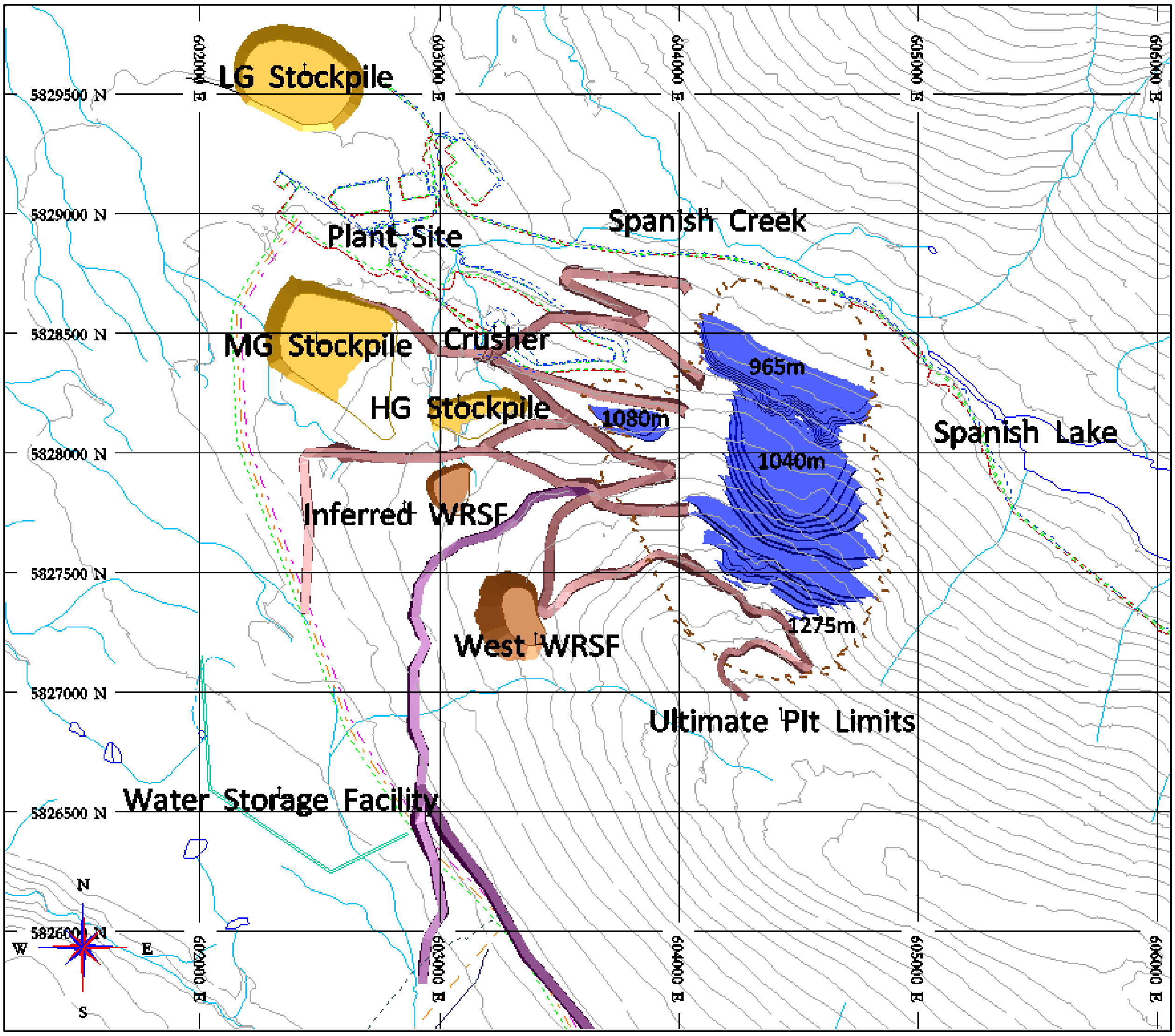


Figure 16-19 Year 2 End of Period Map

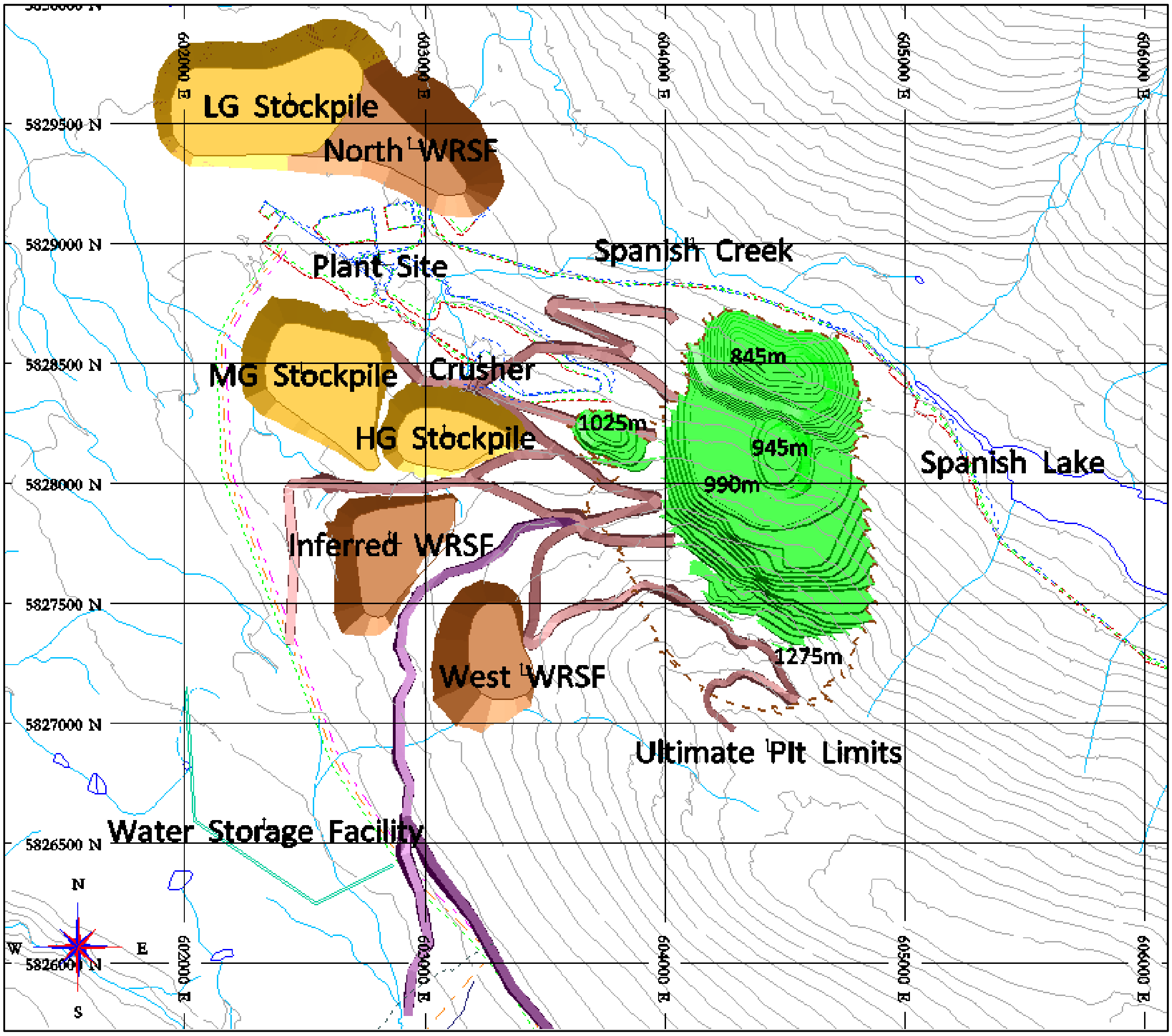


Figure 16-20 Year 5 End of Period Map

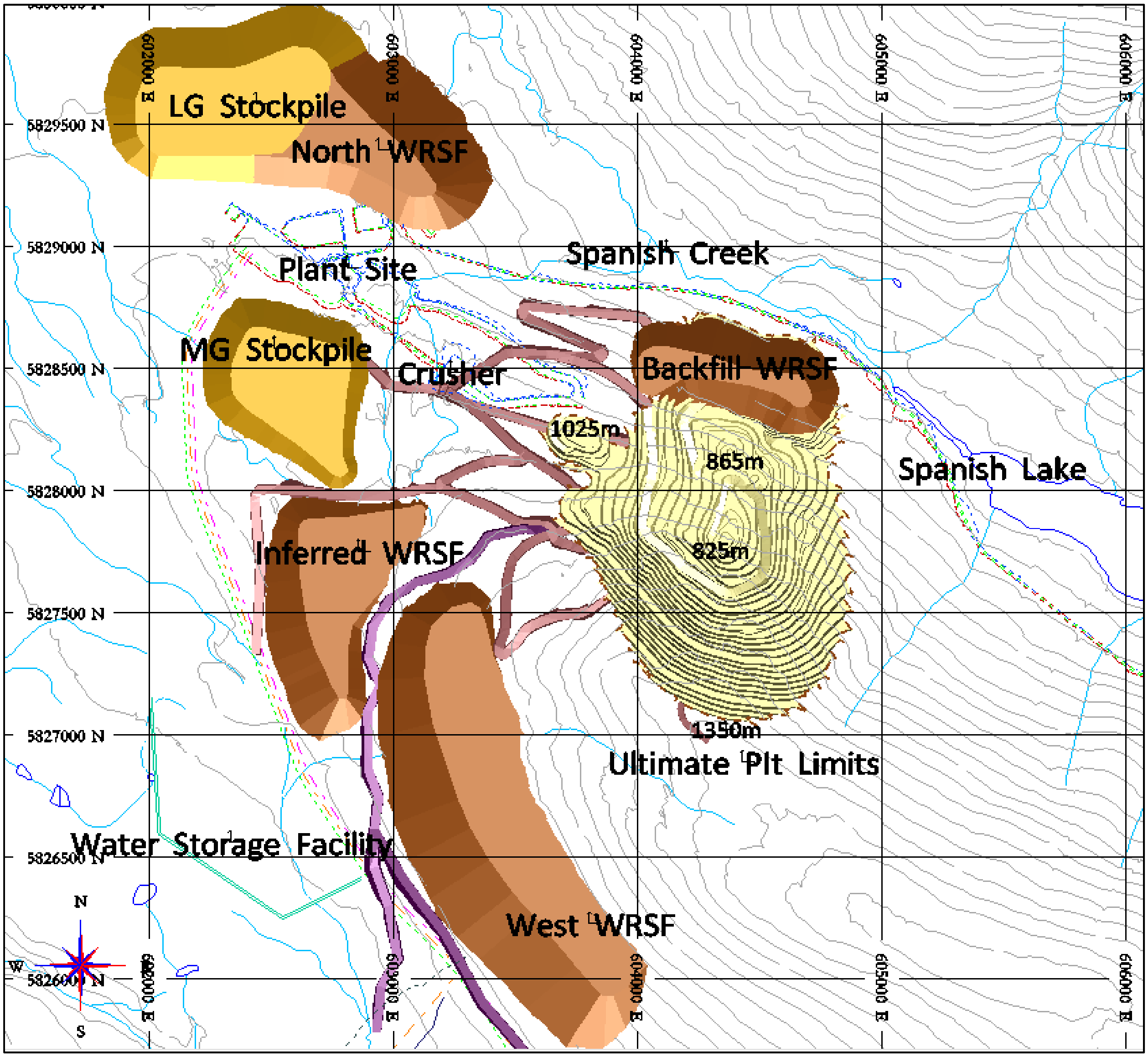


Figure 16-21 **Year 12 End of Period Map**

17.0 Recovery Methods

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 design produces gold-silver doré as a final product.

The 20,000 t/d process plant flowsheet design uses a conventional process technology. Comminution includes crushing, semi-autogenous grinding with pebble crushing, ball mill grinding, and cyclone classification. Cyclone overflow reports to flotation to produce a sulphide concentrate containing the precious metals. The flotation circuit includes a rougher stage with two open-circuit cleaner stages. A scavenging gravity concentration circuit for the cleaner and recleaner flotation tailings minimizes potential gold losses from the open flotation circuit. Flotation tailings are pumped to the tailings impoundment area for storage. Flotation concentrate is thickened and finely ground before leaching in the CIL circuit. Loaded carbon is transferred from the head CIL tank to an elution circuit. Loaded carbon is acid-washed to remove calcium and other impurities followed by the gold elution process. Gold is recovered from the elution solution by electrowinning. Eluted carbon is regenerated in a kiln prior to screening for the removal of carbon fines. Screened regenerated carbon is subsequently returned to the CIL. CIL tailings will be pumped to cyanide detoxification tank where cyanide levels are chemically reduced to acceptable environmental levels prior to disposal to the TMF, separate from the rougher tailings.

Process water is recycled from the flotation concentrate thickener overflow and supplemented with process water recovered from the TMF and water management pond.

The simplified flowsheet is shown in Figure 17-1.

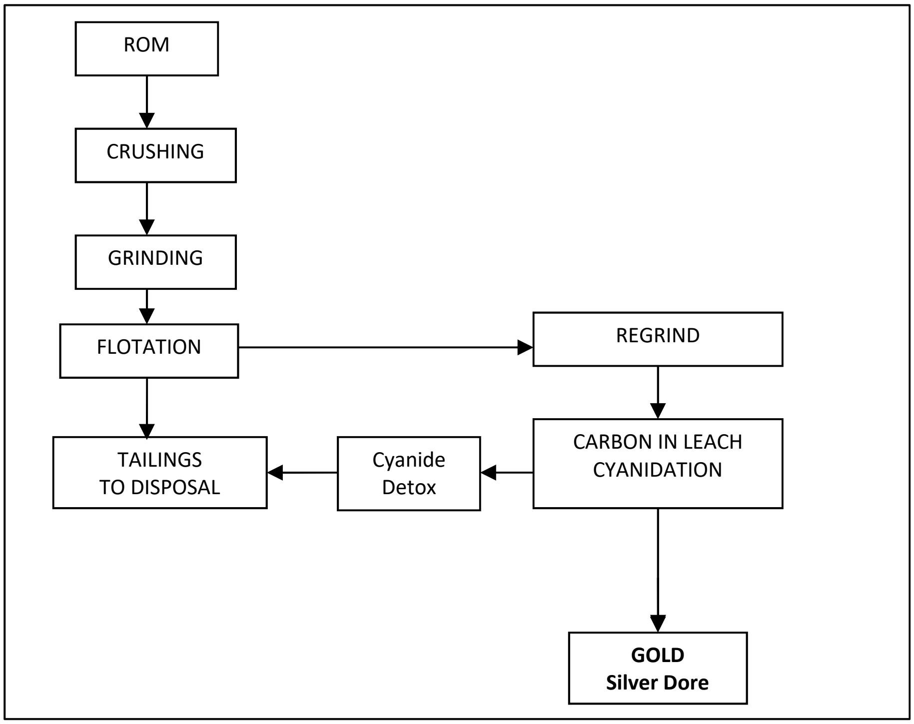


Figure 17-1 Simplified Process Flowsheet for 20,000 t/d

A circuit design utilizing staged flotation reactors has been examined but not incorporated into the PEA process design for Spanish Mountain Gold. This opportunity to decrease project capital and operating costs is discussed in Appendix 17-1. It is recommended that specific test work required for the staged flotation reactors be conducted as part of the next level of project design.

17.1 Major Design Criteria

The concentrator has been designed to treat gold-bearing material at the rate of 20,000 t/d. The major design criteria are outlined in Table 17-1.

Table 17-1 Major Design Criteria

Criteria	Unit	Value
Overall Plant Feed	t/d	20,000
Operating Year	d	365
Primary Crushing Circuit Utilization	%	80
Grinding, CIL and Carbon Circuits Utilization	%	92
Bond Ball Mill Work Index, design	kWh/t	13.3
Bond Abrasion Index, Design	g	0.269
Specific Gravity Feed	-	2.76
SAG Mill Feed Size, 80% Passing	µm	150,000
Ball Mill Feed Size, 80% Passing	µm	1,750
Ball Mill Product Size, 80% Passing	µm	184
Ball Mill Circulating Load	%	250
Regrind Mill Product Size, 80% Passing	µm	20
Rougher float mass pull	%	6
Cleaner float mass pull	%	3
Pre-aeration retention Time	h	12
Leach Circuit Retention Time	h	48

17.2 Operating Schedule and Availability

The crushing and processing plants will be designed to operate two 12-hour shifts per day, for 365 d/a.

The primary crusher operational utilization will be 80% and the grinding, flotation, CIL, and carbon circuit utilization will be 92%. These utilizations will allow for a potential increase in crushing rate, and will allow sufficient downtime for the scheduled and unscheduled maintenance of the crushing and process plant equipment.

17.3 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:

- fire assay equipment
- atomic absorption spectrophotometer (AAS)
- x-ray fluorescence spectrometer (XRF)
- 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 Management Facility (TMF) and water management system.

18.1 On-Site Infrastructure

On-site infrastructure includes:

- Electrical Substation
- Tailing Management 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 include workshops, maintenance shop, warehousing and security. It is assumed in future study designs, several of these services will be located within shared buildings.

A high-level location 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. 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 typically house a wash bay complete with pressure water, 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 overhead cranes.

The wash bay will include sumps with grates and truck washings will be collected 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 Spanish Mountain 1300 Forest Service Road (FSR). Property is accessed from the Spanish Mountain 1300 FSR. This road currently travels through the proposed mine site; it will require rerouting in order 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.

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 TMF, 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 WRSF's, and the TMF 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, described in Section 17.3.

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 5^0 C during the cold season, by providing 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 18 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 from the tailings management facility and water management pond 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 TMF is to provide secure containment of all tailings solids and PAG/ML waste rock.

The tailings streams, rougher and cleaner, will be transported from the plant site to the TMF in separate pipelines. Each tailings stream will be deposited independently; the rougher tailings will be discharged along the TMF 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 TMF 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 TMF will comprise a north embankment and a south embankment. The embankments will be zoned earthfill/rockfill structures, with a low-permeability core for seepage management. The embankments will also 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 material.

The starter TMF—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 TMF 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 TMF will be approximately 178 Mt of tailings, 32 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 TMF 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 TMF from a series of large diameter valved off-takes located along the embankments. Selective tailings deposition will be used to keep the tailings pond away from the embankments to reduce seepage losses from the TMF and encapsulate the cleaner tailings.

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 via an HDPE pipe from a floating barge on the TMF to the water management pond. The barge will be positioned at the north end of the TMF to minimize pumping distance to the water management pond. The reclaim water will subsequently be pumped via an HDPE pipe 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 component, providing a buffering for process water, direct precipitation and runoff.

Surface diversion ditches will capture and divert non-contact water around the TMF for release to the environment. Runoff from catchments directly upstream of the TMF 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 TMF.

18.5 Waste Rock Management

18.5.1 Waste Rock Production

MMTS developed a production schedule which defined the amount of material produced annually over the mine life. The waste rock from this schedule is identified and categorized based on its geochemical characterization (see Section 16.9).

18.5.2 Waste Rock Disposal Strategy

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

The remainder of the waste rock will be disposed of in the West, North, Inferred and Backfill WRSF's shown in Figure 1-3.

18.6 Off-Site Infrastructure

The Project requires roughly 20 MW of peak load for 20,000 t/d operation demand. The power will be supplied by a new transmission line interconnecting the SMG site to BC Hydro's power system.

BC Hydro conducted a system impact study (SIS) for a 60 MW peak load line, which determined the most suitable point of interconnection to BC Hydro's grid, and the estimated costs associated with system reinforcement as follows (from BCHydro, 2013):

"Various alternative Points of Interconnection and voltage levels were studied. After several discussions with the customer involving various issues, it was identified that the technically leading alternative to interconnect the customer's load is via a new 230 kV substation (MCS), on 2L95, approximately 15 km from Soda Creek, in the vicinity of the McLeese 500 kV series capacitor station.

BC Hydro conducted this SIS to identify the method of interconnection and any major transmission facilities required to supply the requested load, including associated protection, control and telecommunication requirements."

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$1,050 and US$1,360 per ounce. The average gold price since June 2014 is US$1,270 per ounce. A gold price of US$1,250 per ounce, and a silver price of US$18 per ounce, are considered reasonable with respect to the current market and have been used for this assessment.

SMG 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.

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

20.2 Waste Rock and Tailings Disposal, Site Monitoring, and Water Management

Waste rock and tailings disposal, and their attendant water management strategies are discussed in Section 18.0.

The Spanish Mountain resource has a low potential for ML or ARD, especially if waste rock segregation strategies can be incorporated into proposed mining methods.

Characterization work has been conducted with laboratory humidity cells and on-site field (barrel) tests initiated for kinetic evaluation of ML/ARD potential. Metallurgical process wastes were also being evaluated.

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 management 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 TMF. 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

A typical EA is generally completed in three to four years. For this Project, the EA process began on July 8, 2011, with the submission of a project description to the BC Environmental Assessment Office (EAO) and the federal Canadian Environmental Assessment Agency (CEAA). Detailed environmental and socio-economic baseline studies were then initiated; these typically require a two-year period to complete. 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.

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

When this EA review is complete, a provincial EA certificate and federal Notice of Decision will be issued. 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 govern the participation of each party during the EA and permitting review of the Project.

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 (WRSF's 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.

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 Q1 2017 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 TMF (including the associated tailing delivery and return pipelines to and from the process plant) and Water Management.
- 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.

Further details of the Basis of Estimate can be found in Appendix 21-1.

Table 21-1 Capital Cost Summary

Direct Costs		Initial Capital Cost (M$)
	Site Control and Communications	2.1
	Site Preparation	6.7
	Site Electrical	7.8
Overall Site Subtotal		16.6
	Pre-Production Operations (including TMF bulk earthworks)	33.3
	Mining Equipment	57.0
	Explosives Facilities	1.5
	Mining Dewatering	4.0
	Communication Systems	1.5
Open Pit Mining Subtotal		97.3
Processing Plant (including Ore Handling) Subtotal		140.0
	Tailings and Water Management Earthworks	39.8
	Reclaim Pumping	2.1
	Water Treatment Facility	15.0
Tailing Management Facility & Water Management Subtotal		56.8
Environmental Subtotal		12.0
	Ancillary Buildings	21.1
	Site Services and Utilities	3.4
	Plant and Site Mobile Equipment	4.1
On-Site Infrastructure Subtotal		28.6
Off-Site Infrastructure Subtotal		14.3
Sub-Total Direct Costs		**365.6**
Indirect Costs		
	Project Indirects	84.6
	Owner's Costs	5.8
	Contingencies	51.1
Sub-Total Indirect costs		**141.5**
Total Initial Capital Cost		**507.1**

21.2 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.

The following table shows all initial and sustaining capital cost estimates for the project, in $M.

Table 21-2 Initial and Sustaining Capital Cost Estimates

Year	LOM	Initial	Sustaining	-2	-1	1	2	3	4	5	Y6-10	Y11-15	Y16-20	Y21-25
DIRECT COSTS:														
Site		$16.6	$0.0	$6.6	$10.0									
Mining		$97.3	$85.5	$38.9	$58.4	$11.8	$1.5	$52.8	$0.1	$0.2	$16.1	$2.6	$0.4	
Processing		$140.0	$0.0	$56.0	$84.0									
Tailings Dam and Water Management		$56.8	$91.0	$22.7	$34.1	$3.0	$4.8	$4.8	$4.8	$4.8	$23.8	$17.8	$13.8	$13.8
Environmental		$12.0	$0.0	$4.8	$7.2									
Site Infrastructure		$42.9	$0.0	$17.1	$25.7									
Total Direct Costs	**$542.1**	**$365.6**	**$176.5**	**$146.2**	**$219.3**	**$14.8**	**$6.3**	**$57.6**	**$4.9**	**$5.0**	**$39.9**	**$20.4**	**$14.2**	**$13.8**
INDIRECT COSTS:														
Indirect Costs		$84.6	$0.0	$33.8	$50.8									
Owners Costs		$5.8	$0.0	$2.3	$3.5									
Reclamation Costs		$0.0	$32.0									$1.0	$5.0	$26.0
Salvage Values		$0.0	-$15.0									-$5.0		-$10.0
Total Indirect Costs	**$107.4**	**$90.4**	**$17.0**	**$36.2**	**$54.2**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**-$4.0**	**$5.0**	**$16.0**
CONTINGENCY COSTS:														
Contingency		$51.1	$0.0	$20.4	$30.7									
Total Contingency	**$51.1**	**$51.1**	**$0.0**	**$20.4**	**$30.7**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**
TOTAL COSTS:														
Total Capital		$507.1	$193.5	$202.8	$304.2	$14.8	$6.3	$57.6	$4.9	$5.0	$39.9	$16.4	$19.2	$29.8
Total Project Capital	**$700.6**	**$507.1**	**$193.5**	**$202.8**	**$304.2**	**$14.8**	**$6.3**	**$57.6**	**$4.9**	**$5.0**	**$39.9**	**$16.4**	**$19.2**	**$29.8**

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

An initial CAPEX was completed for site development, on-site and off-site infrastructure. Section 18.0 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.

21.2.2 Mining

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

- Pre-production cost is estimated for 8.3 Mt of waste and 3.0 Mt of mineralized material mined in pre-production. The costs include hauling 5.7 Mt of suitable waste rock to the tailings 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 is 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 haulroads to be undertaken by a contractor.
- Salaries and costs for mine and engineering staff, and consultants during the pre-production period are included.
- The cost for the truck dispatch system is included with this estimate. Other capital cost allowances for 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.

21.2.3 **Processing**

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

- Crushing
- Grinding
- Flotation
- Scavenger Gravity
- Cyanidation (CIL)
- Tailings Delivery
- Reagents and Consumables
- Plant Services
- Tailings Storage
- 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 40,00 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.

21.2.4 Tailings Dam 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 TMF embankment footprints, laydown areas, topsoil and unsuitables 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 TMF 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.
- TMF embankment monitoring instrumentation (piezometers and inclinometers).
- Sustaining capital covering the tailings and reclaim system operating costs, as well as operating costs of the water treatment plant.
- 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.

21.2.5 Environmental

Habitat compensation costs for the TMF 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 TMF 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 TMF 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.

21.2.6 **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.

Reclamation bonding has not been included in the capital cost estimate.

21.2.7 **Owner's Costs**

The Owner's costs are estimated to be $5.8 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
- Administration
- Facilities Services
- 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
- Housing Costs
- Owner's Contingency

21.2.8 Reclamation and Salvage Values

Sustaining CAPEX of $32.0 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)	$8,000
WRSF and Tailings Faces and Tailings Tops ($/ha)	$40,000
Roads and Berms ($/ha)	$5,000
Infrastructure ($/ha)	$4,000

Progressive reclamation is planned between Year 15 and Year 25, at $1,000,000/year, with the balance applied in Year 25. Salvage values are split up and applied at the end of pit mining in Year 14 ($5.0 million), and at the end of mine life in Year 25 ($10. million).

21.2.9 Contingency

The overall contingency for the Project is $51.1 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.

21.3 Operating Cost Estimate

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

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

Table 21-4 Unit Operating Costs

Area	Unit Cost
Mining ($/t mined)	$1.96
Mining ($/t milled)	$4.79
Processing ($/t milled)	$4.01
Tailings ($/t milled)	$0.05
G&A ($/t milled)	$1.09
Total ($/t milled)	**$9.92**

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

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

Unit Production Costs per ounce	First 5-Yrs	First 10-Yrs	Life of Mine
Cash Cost	$626	$740	$793
All-in-Sustaining Cost (AISC)	$711	$825	$878
Total Cost	$889	$1,003	$1,057

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)	$0.80
Power ($/kWh)	$0.060

21.3.1 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	$0.09	$0.23
Blasting	$0.20	$0.49
Loading	$0.34	$0.84
Hauling	$0.80	$1.96
Pit Support	$0.25	$0.60
Geotechnical	$0.03	$0.08
Unallocated Labour	$0.02	$0.05
DIRECT COSTS - Subtotals	$1.73	$4.25
Mine Operation/Maintenance GME	$0.11	$0.27
Mine Engineering GME	$0.11	$0.27
TOTAL GME COSTS	$0.23	$0.54
Total Operating Cost	**$1.96**	**$4.79**

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 hauler productivities range from 420 to 700 tonnes/operating hour.

21.3.2 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	$0.88
Consumables	$1.59
Power	$1.30
Maintenance	$0.23
Total	**$4.01**

21.3.3 **Tailings Operating Costs**

Tailings and water treatment operating costs have been mostly included as sustaining capital, under the "Tailings Dam and Water Management" label. An additional \$0.05/t milled have been added to cover additions and operations of the tailings distribution and embankment seepage collection and recycle systems.

21.3.4 **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 \$8.0 million, or \$1.09/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
- Safety Incentives
- 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

In line with the level of the PEA study, by NI43-101 standards, it is estimated that the data used, and the conclusions reached for the economic analysis are within ±40% for this report.

Spanish Mountain Gold's taxation model, as of Q1 2017, has been used to estimate federal, provincial, and other taxes. Some additional details are included in Section 0.

All dollar amounts in this analysis are expressed in Q1 2017 Canadian dollars, unless specified otherwise.

The economic analysis is run over the entire project life, comprising two years of construction and 24 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 determining the project's taxable income. The IRR assumes 100% equity financing.

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	$1,250	US$/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***	$1.96	$/t mined
Gold Process Recovery (Y1 to 11)	90%	
Gold Process Recovery (Y12 to 24)	87%	
Silver Process Recovery	40%	
Processing Costs	$4.01	$/t milled
General & Administration Costs	$1.09	$/t milled
TMF Operating Costs	$0.05	$/t milled

** % of Net Value after smelter charges have been applied*
*** It is anticipated that NSR obligations under the 'Wallster and McMillan' and 'R.E. Mickle' claims, described in Section 4.3, will be purchased by the owner in advance of commercial production, lowering the overall NSR commitment within the delineated resource to 1.5%.*
**** Variable annual mining costs based on scheduled open pit production, LOM average of $1.96/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	178	Mt
Au Grade	0.43	g/t
Au Produced	2,210	koz
Ag Grade	0.67	g/t
Ag Produced	1,535	koz
Waste Mined	257	Mt
Strip Ratio	1.4	t:t
Initial Capital	507	$M
Sustaining Capital	194	$M
Cash Cost	595	$/oz
Net Cash Flow	963	$M
Pre-Tax (SMG)		
NPV, 5%	597	$M
IRR	21%	%
Payback	3.7	Years
Post-Tax (SMG)		
NPV, 5%	482	$M
IRR	19%	%
Payback	3.7	Years

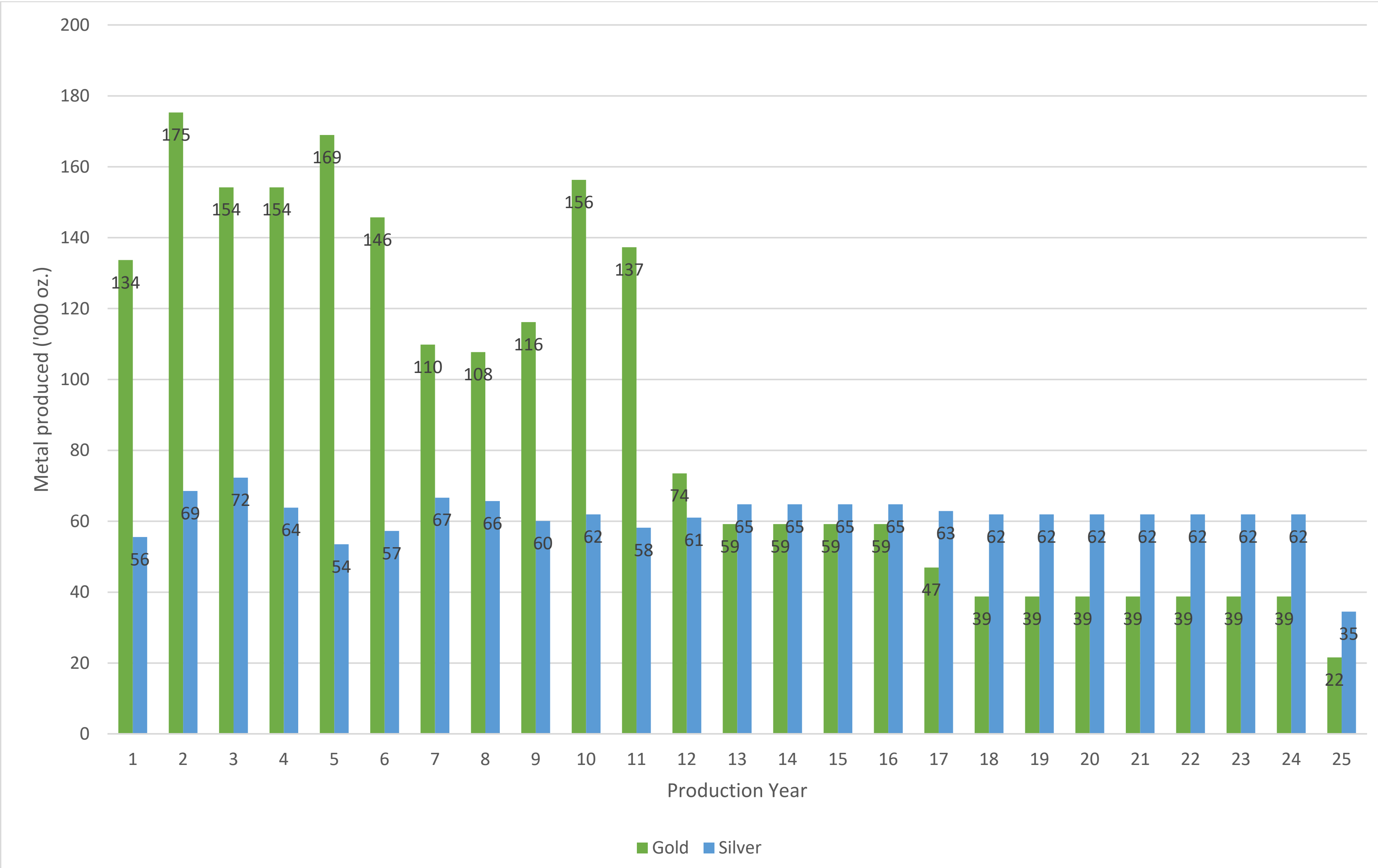
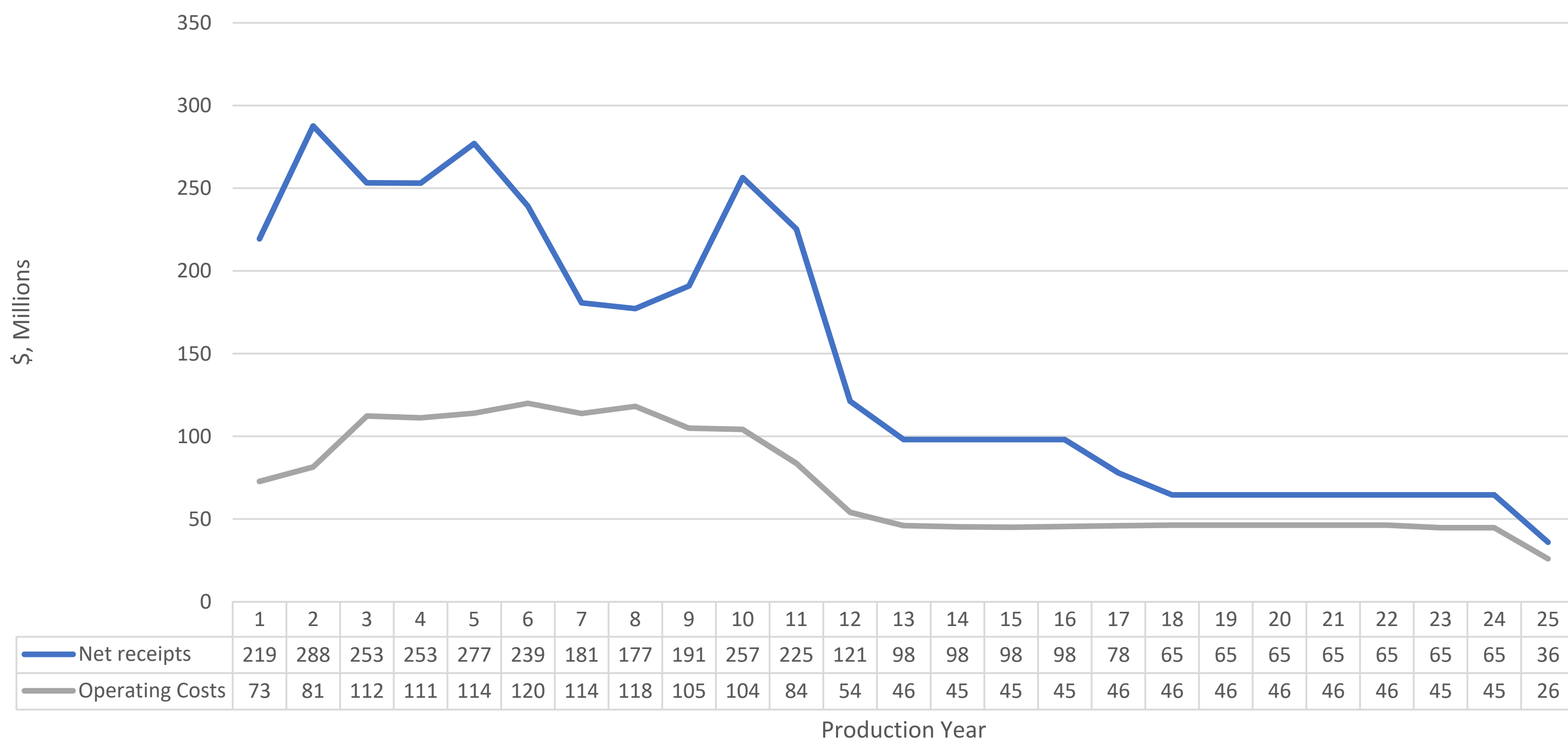


Figure 22-1 LOM Gold and Silver Production

The following graph, Figure 22-2, shows by year:

- the estimated net gold and silver receipts
    - o gross gold and silver receipts minus offsite charges: refining, transport, insurance and royalty charges
- the estimated operating costs
    - o mining, processing, TMF and G&A costs

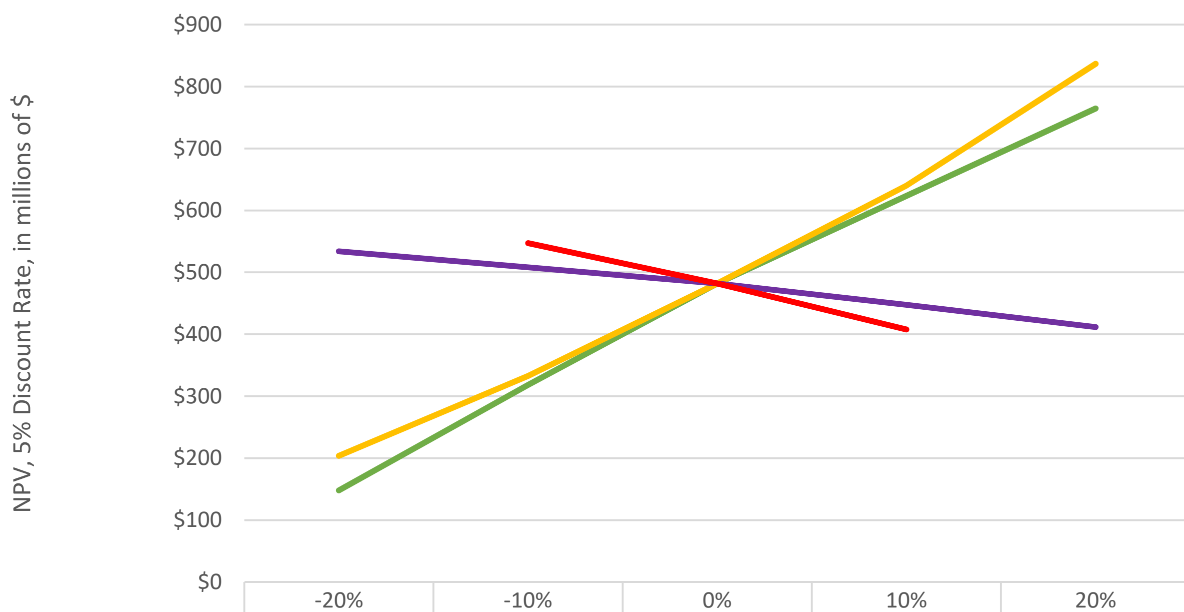


Production Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25
Net receipts	219	288	253	253	277	239	181	177	191	257	225	121	98	98	98	98	78	65	65	65	65	65	65	65	36
Operating Costs	73	81	112	111	114	120	114	118	105	104	84	54	46	45	45	45	46	46	46	46	46	46	45	45	26

Figure 22-2 Net Receipts vs. Operating Costs

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

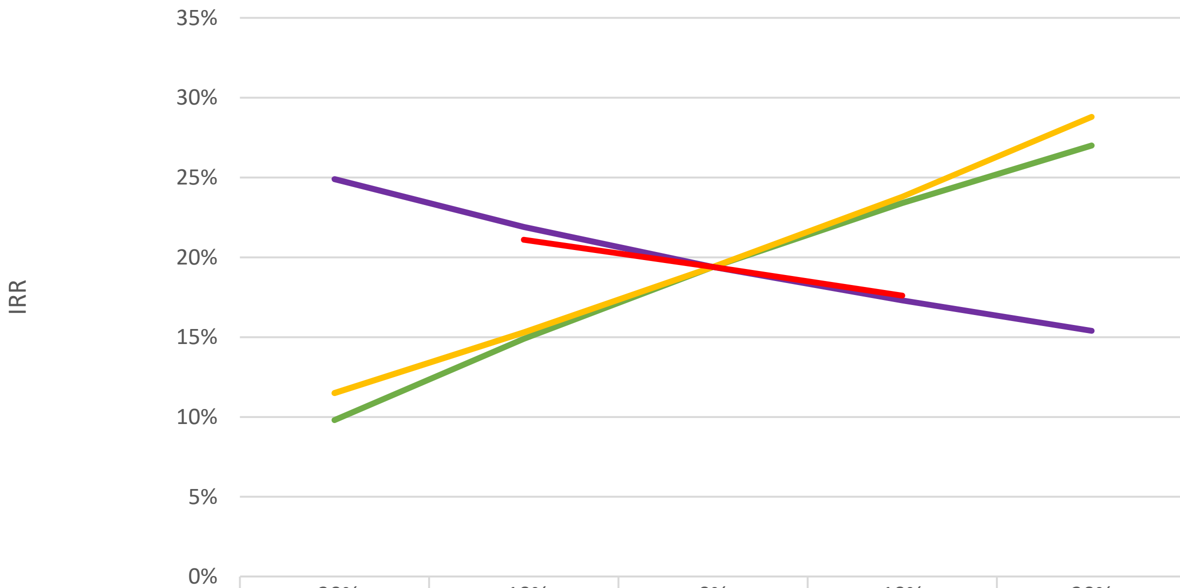
- Gold Price
- Foreign Exchange Rate
- Project Capital Costs and
- Operating Costs (mining, processing, TMF and G&A costs)



	-20%	-10%	0%	10%	20%
Gold Price	148	318	482	624	765
Exchange Rate	204	333	482	640	837
Capital Costs	534	508	482	448	412
Operating Costs		547	482	408	

Parameter Variance

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



	-20%	-10%	0%	10%	20%
Gold Price	10%	15%	19%	23%	27%
Exchange Rate	12%	15%	19%	24%	29%
Capital Costs	25%	22%	19%	17%	15%
Operating Costs		21%	19%	18%	

Parameter Variance

Figure 22-4 Sensitivity of Post-Tax IRR to various project inputs

22.1 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 24-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)	101.1
Corporate Tax (Provincial)	74.1
Less Investment Tax Credit (ITC)	(2.1)
Provincial Resource Tax	37.5
Total Taxes	**210.6**

The following general tax regime was recognized as applicable at the time of report writing.

22.1.1 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 11% 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.

22.1.2 BC Mineral Tax Regime

The BC Mineral Tax regime is a two-tier tax regime, with a 2% tax and a 13% 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

The Property is in an area that has seen active past exploration and mining activity for alkaline porphyry copper-gold deposits. Currently, the most advanced property in the area is Imperial Metals' Mount Polley Mine, which is an alkalic porphyry copper-gold deposit located about 15km to the west. As of May, 2016, the deposit has proven and probable reserves of 73.6 million tonnes grading 0.27% copper and 0.29 g/t gold, as well as Measured and Indicated resources of 247 million tonnes grading 0.27% copper and 0.26 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.

24.0 Other Relevant Data and Information

No additional relevant information or data to disclose.

25.0 Interpretation and Conclusions

A PEA open pit mine plan has been developed using NI 43-101 Resource estimates. The PEA shows positive economic viability.

25.1 Resource Conclusions

The mineral resource is suitable for a PEA study.

- SMG has been drilling on the Property since 2005. To date SMG drilling totals about 190,000 m in more than 740 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, and are NI 43-101 compliant.
- In total, 669 core drillholes (153,828 m) from 2005 to 2012 inclusive and 145 RC drillholes (19,034 m) from 2004 to 2006 and 2013-2014 have been used to determine the Resource.
- The mineral resource contains 46 Mt of 0.53 g/t Au and 0.66 g/t Ag in the Measured category; 261 Mt of 0.35 g/t Au and 0.67 g/t Ag in the Indicated category; and 451 Mt of 0.28 g/t Au and 0.0.61 g/t Ag in the Inferred category, based on a 0.15 g/t Au cut-off.
- Of the 307 Mt classified as Measured and Indicated, a total of 178 Mt or 58% are within the delineated open pit designs for this PEA.

25.2 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 7.3 Mt of resource per year over a 24-year period.
- The mine plan supports the cash flow model and financials developed for the PEA.
- The open-pit resources 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 six mineable phases, or pushbacks, to target higher grade resource earlier in the project.
- There is a total pit delineated resource of 178 Mt at an average gold grade of 0.44 g/t, an average silver grade of 0.67 g/t, and a waste to resource strip ratio of 1.4.
- Pit delineated resources are run using a 0.15 g/t gold grade cut-off, which covers process, G&A, and stockpile to plant re-handle costs.
- 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 7.3 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.

25.3 Process Conclusions

The evaluation of process options indicates that:

- Gold mineralization is fine-grained particles requiring concentrate regrind prior to leaching.
- Gold is predominantly associated with quartz and sulphide minerals (mainly pyrite).
- Preg-robbing organic carbon is successfully removed from flotation concentrate using CMC as a suppressant.
- Overall LOM gold recovery of 89%, and silver recovery of 40%, is achievable.
- The process plant is based on a 20,000 t/d throughput and a flowsheet design including crushing, grinding, multistage flotation, scavenger gravity of cleaner tails, concentrate regrind, and CIL to produce doré.
- Two tailings streams are produced: rougher tailings and cleaner/CIL tailings, which will be transported from the plant site to the TMF in separate pipelines. Each tailings stream will be deposited independently; the rougher tailings will be discharged along the TMF 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 TMF capacity will be approximately 178 Mt of tailings, 32 Mt of PAG/ML waste rock, plus the supernatant pond volume and freeboard allowances.
- The TMF embankments will be constructed using suitable waste rock and overburden (low permeability glacial till) from the open pit.

26.0 Recommendations

The positive conclusions of this PEA lead the authors to recommend that the Project should proceed towards a higher level of 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 staged approach is eventually directed towards the completion of a FS on the Spanish Mountain Gold Project.

Table 26-1 Recommended future studies

Item	Estimated Budget (M$)
Infill Drilling and Resource Update	$1.0
Condemnation Drilling	$1.3
Geotechnical Drilling and Studies	$1.8
Environmental Testing	$1.5
Site Studies (TSF, Facilities, BCHydro, Pumping, Seismic, Metallurgy, Mining, AIA)	$3.0

The estimated dollar amount for these items are 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 GCL recommend that future studies consider the following elements:

- Additional infill drilling to re-classify the material within the open pit currently defined as an inferred resource.
- For any future drill programs, it is recommended that RC drilling be utilized.

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.).

26.3 Process Recommendations

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

- Additional metallurgical testwork using samples from new drill core to finalize the process flowsheet, develop recovery projections, mass balance, and design assumptions.
- Complete preliminary process engineering and plant design.
- Specific test work required for the staged flotation reactors should be conducted.

26.4 Infrastructure and Tailings Recommendations

MMTS and KP recommends 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 AIA studies.
- Initiate engineering studies for the water balance, water quality and water management on site.
- A surficial geology study and geotechnical site investigation in the TMF 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.
  - The site investigation program would include geochemical and hydrogeological drilling to support environmental baseline studies, test pitting, seismic surveys, laboratory test work and analysis.

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

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 (BCHydro, 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

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

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

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

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

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

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): Preliminary Mining Study on Spanish Mountain, December 18, 2012

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

Page, J.W. (2003): Compilation Report: A summary of the Exploration Programs and Results, Spanish Mountain Property, British Columbia, for Wildrose Resources Ltd; 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

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

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

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

28.0 Certificates

CERTIFICATE & DATE – Marc Schulte, P.Eng.

I, Marc Paul Schulte, P.Eng., of Edmonton, Alberta do hereby certify that:

1) I am a Mining Engineer, with Moose Mountain Technical Services, with a business address of #210 1510 2nd St North Cranbrook BC, V1C 3L2.
2) I graduated with a Bachelor of Science in Mining Engineering from the University of Alberta in 2002.
3) I am a member of the Association of Professional Engineers, Geologist and Geophysicists of Alberta. (#71051).
4) I have worked as a Mining Engineer for a total of 15 years since my graduation from university.
5) I have worked on base metal and coal mining projects in western Canada, including mine operations and mine evaluations for 17 years. I have experience in base metal mining operations and preparing project evaluations for gold deposits in Canada.
6) I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with professional associations (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
7) I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
8) I am independent of Spanish Mountain Gold in accordance with Section 1.5 of NI 43-101.
9) I have prepared and am responsible for Sections 1, 2, 3, 15, 16, 18, 19, 21, 22, 24 and portions of Sections 25, 26 and 27 of the report titled "**Preliminary Economic Assessment of the Spanish Mountain Gold Project**" dated 17 May 2017 (the 'Technical Report").
10) I have visited the Property in October 2011.
11) To the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
12) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 17th day of May 2017

{Signed and Sealed}

____

Signature of Qualified Person
Marc Schulte, P.Eng.

CERTIFICATE & DATE – William Gilmour, PGeo

I, William Gilmour, of Coldstream, British Columbia, do hereby certify that:

1) I am a Geologist with Discovery Consultants, with a business address of 2916, 29th Street, Vernon, BC, V1T 5A6.
2) I graduated with a Bachelor of Science in Geology from the University of British Columbia in 1970.
3) I am a member of the Association of Professional Engineers and Geoscientists of British Columbia (membership #19743).
4) 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.
5) On the Spanish Mountain Gold Project, I have monitored the analytical results, including quality control and quality assurance analyses, for the 2012, 2013 and 2014 drill programs.
6) I have read the definition of “qualified person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with professional associations (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
7) I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
8) I am independent of Spanish Mountain Gold in accordance with Section 1.5 of NI 43-101.
9) I have prepared and am responsible for Sections 4 through 12, 23 and portions of Sections 25, 26 and 27 of the report titled “**Preliminary Economic Assessment of the Spanish Mountain Gold Project**” dated May 17, 2017 (the “Technical Report”).
10) I visited the Property during the drill programs in 2012 and 2013.
11) To the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
12) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 17th day of May 2017

{Signed and Sealed}

____
Signature of Qualified Person
William Gilmour, PGeo

CERTIFICATE & DATE – Gary H. Giroux, P.Eng.

I, Gary H. Giroux, P.Eng., of North Vancouver, British Columbia do hereby certify that:

1) I am a Consulting Geological Engineer with Giroux Consultants Ltd., with a business address of 982 Broadview Drive, North Vancouver BC, V7H 2G1.
2) This certificate applies to the technical report entitled titled "**Preliminary Economic Assessment of the Spanish Mountain Gold Project**" dated 17 May 2017 (the 'Technical Report").
3) I am a graduate of the University of British Columbia, (B.A.Sc. in Geological Engineering, 1970 and M.A.Sc. in Geological Engineering, 1984).
4) I am a member in good standing of the Association of Professional Engineers, and Geoscientists of British Columbia, License #8814.
5) I have practiced my profession continuously since 1970. I have had over 40 years of experience estimating mineral resources. I have completed numerous resource estimations on bulk tonnage gold deposits such as La India, Livengood and Sleeper.
6) I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with professional associations (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
7) I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
8) I am independent of Spanish Mountain Gold in accordance with Section 1.5 of NI 43-101.
9) I have prepared and am responsible for Section 14 and portions of Sections 25, 26 and 27 of the Technical Report.
10) I have last visited the Property in June 29, 2011.
11) I have prior involvement with the Property that is the subject of the Technical Report. I have completed resource estimates in 2008, 2011 and co-authored the August 2012 Technical Report.
12) To the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
13) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 17th day of May 2017

{Signed and Sealed}

____

G. H. Giroux, P.Eng.

CERTIFICATE & DATE – Les Galbraith, P.Eng.

I, Les Galbraith, P.Eng., of Vancouver, British Columbia do hereby certify that:

1) I am a Specialist Engineer/ Project Manager with Knight Piésold Ltd. with a business address at Suite 1400 – 750 West Pender Street, Vancouver, British Columbia, V6C 2T8.
2) This certificate applies to the technical report entitled Preliminary Economic Assessment of the Spanish Mountain Gold Project, Likely, BC, dated May 17, 2017 (the "Technical Report").
3) I am a graduate of the University of British Columbia, (B.A.Sc., 1995). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #25493. My relevant experience is 22 years of experience in providing civil and geotechnical engineering support to mining and hydroelectric projects.
4) I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with professional associations (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5) My most recent personal inspection of the Property was October 4, 2012.
6) I am responsible for Sections 18.4, 20, and portions of Sections 25, 26, and 27 of the Technical Report.
7) I am independent of Spanish Mountain Gold Ltd. as defined by Section 1.5 of the Instrument.
8) I have no prior involvement with the Property that is the subject of the Technical Report.
9) I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
10) As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all of the scientific and technical information that is required to be disclosed to make the technical report not misleading.
11) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public.

Signed and dated this 3rd day of May, 2017 at Vancouver, British Columbia

{Signed and Sealed}

____

Les Galbraith, P.Eng.
Specialist Engineer | Associate
Knight Piésold Ltd.

CERTIFICATE & DATE – Tracey Meintjes, P.Eng.

I, Tracey Meintjes, P.Eng., of Vancouver B.C. do hereby certify that:

1. I am a Metallurgical Engineer with Moose Mountain Technical Services with a business address at 1975 1st Avenue South, Cranbrook, BC, V1C 6Y3.
2. This certificate applies to the technical report titled "**Preliminary Economic Assessment of the Spanish Mountain Gold Project**" dated 17 May 2017 (the "Technical Report").
3. I am a graduate of the Technikon Witwatersrand, (NHD Extraction Metallurgy – 1996)
4. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#37018).
5. 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.
6. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
7. I have not visited the Property.
8. I am responsible for Sections 13 and 17; including metallurgical and processing portions of Chapters 1, 25 and 26 of the Technical Report.
9. I am independent of Spanish Mountain Gold as defined by Section 1.5 of the Instrument.
10. I have had no previous involvement with the Property that is the subject of the Technical Report.
11. I have read the Instrument and the Technical Report has been prepared in compliance with the Instrument.
12. As of the date of this certificate, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public.

Dated the 17th day of May 2017

{Signed and Sealed}

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Signature of Qualified Person
Tracey D. Meintjes, P.Eng.