NI 43-101 Technical Report - Preliminary Economic Assessment for the Spanish Mountain Project

December 20, 2010

Prepared by

AGP Mining Consultants Inc.

92 Caplan Avenue
Suite 246
Barrie, Ontario, Canada

Spanish Mountain Gold Ltd.

NI 43-101 Technical Report – Preliminary Economic Assessment for the Spanish Mountain Project

Prepared by:

AGP Mining Consultants Inc.

12/20/2010

Contents

Tables

Figures

Appendices

Glossary

Units of Measure

Above mean sea level	amsl
Ampere	A
Annum (year)	a
Centimetre	cm
Cubic centimetre	cm^3
Cubic metre	m^3
Day	d
Days per year (annum)	d/a
Degree	°
Degrees Celsius	°C
Dry metric ton	dmt
Foot	ft
Gram	g
Grams per litre	g/L
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
Horsepower	hp
Hour	h
Hours per day	h/d
Inch	"
Inverse distance	ID
Kilogram	kg
Kilometre	km
Kilovolts	kV
Kilowatt hour	kWh
Kilowatt	kW
Less than	<
Litre	L
Metre	m
Metres above sea level	masl
Metres per minute	m/min
Metres per second	m/sec
Metric ton (tonne)	t
Micrometre (micron)	µm
Millimetre	mm
Million loose cubic metre	Mlm^3

Million tonnes	Mt
Million	M
Minute (plane angle)	'
Minute	min
Nearest neighbour	NN
Newton	N
Newtons per metre	N/m
Ounce	oz
Parts per billion	ppb
Parts per million	ppm
Percent	%
Pound(s)	lb
Second (plane angle)	"
Second (time)	sec
Specific gravity	SG
Square kilometre	km^2
Square metre	m^2
Tonne (1,000 kg)	t
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
Volt	V
Year (annum)	a

Abbreviations and Acronyms

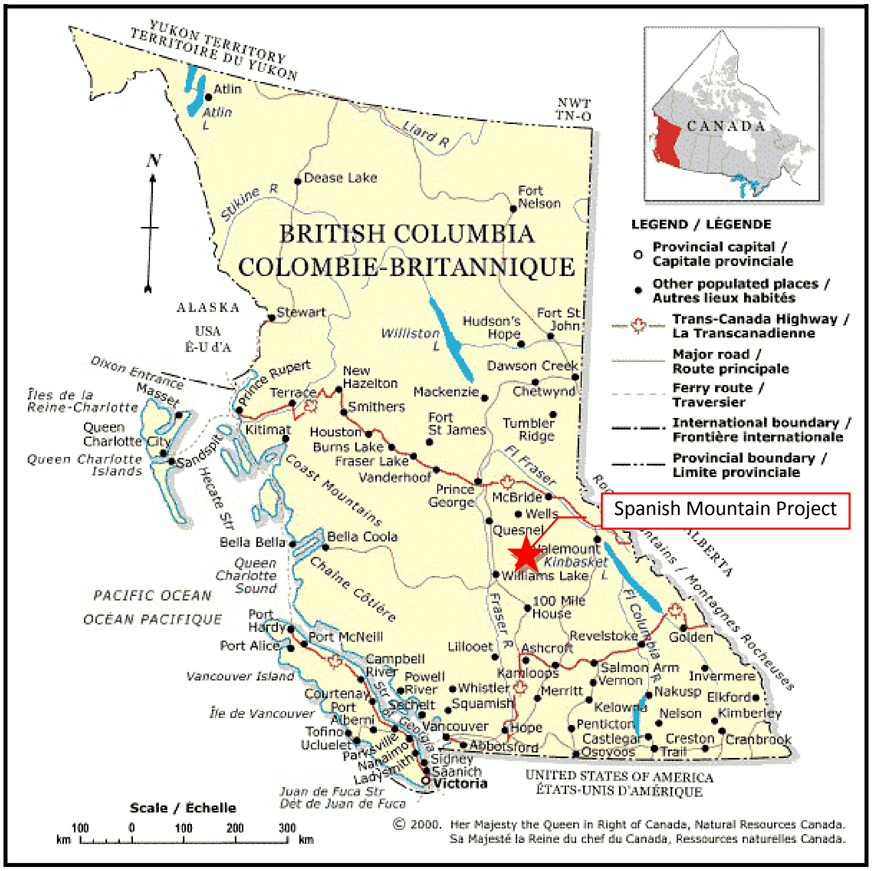
Abrasion Index	Ai
Acid Potential	AP
Acid Rock Drainage	ARD
Acid-base Accounting	ABA
Application for Environmental Assessment Certificate	AEAC
Application Information Requirements	AIR
Ball Work Index	BWi
BC Environmental Assessment Office	EAO
British Columbia Water Quality Guideline	BCWQG
Canadian Council of Ministers of the Environment	CCME
Canadian Environmental Assessment	CEA
Carbon-In-Leach	CIL
Cash Flow Analysis	DCF
Cleaner Scavenger Tailings	CST

Comprehensive Study Report	CSR
Department of Fisheries and Oceans Canada	DFO
Discounted Cash Flow	DCF
EAO Project Information Centre	e-PIC
Engelmann Spruce-Sub-Alpine Fir	ESSF
Environment Canada	EC
Environment Canada	EC
Environmental Assessment	EA
Environmental Assessment Office	EAO
Factors of Safety	FOS
Fisheries and Oceans Canada – DFO Natural Resources Canada	NRC
Interior Cedar–Hemlock	ICH
Knight Piésold	KP
Major Projects Management Office	MPMO
Metal Leaching	ML
Metal Leaching/Acid Rock Drainage	ML/ARD
Metal Mining Effluent Regulations	MMER
Mineral Tenures Online	MTO
Natural Resources Canada	NRC
Natural Resources Canada	NRC
Net Smelter Return	NSR
Neutralization potential	NP
Non-Acid Generating	NAG
Potassium Amyl Xanthate	PAX
Potentially Acid Generating	PAG
Preliminary Economic Assessment	PEA
Project Implementation Plan	PIP
Rock Quality Designation	RQD
Rod Work Index	RWi
Ropes of Gold	ROG
Rougher Scavenger Tailings	RST
Sediment Hosted Vein	SHV
Skygold Ventures Ltd.	Skygold
Spruce-Subalpine Fir	ESSF
Tailings Management Facility	TMF
Wildrose Resources Ltd.	Wildrose

1 SUMMARY

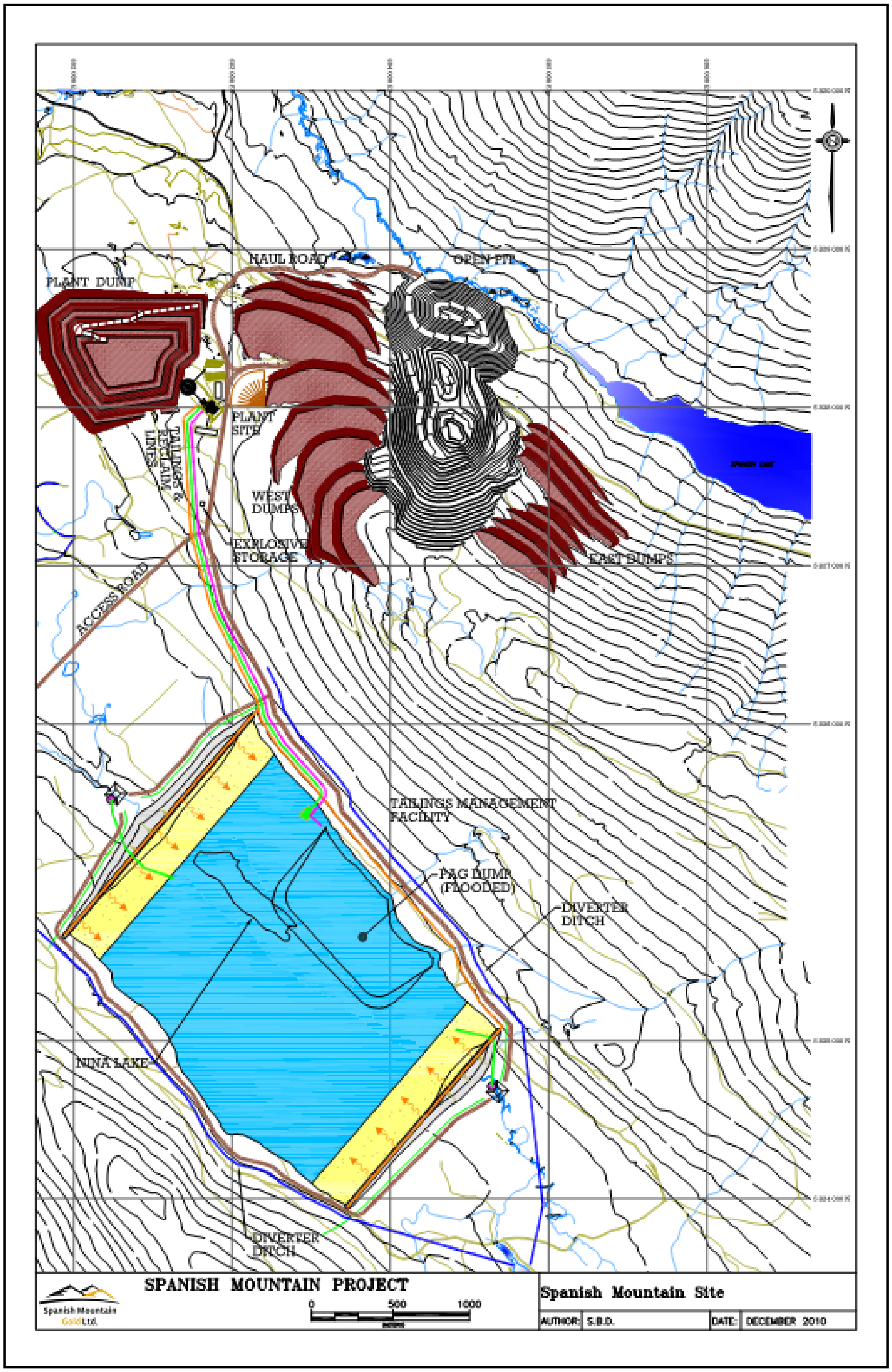
The Spanish Mountain property is located in the Cariboo region of central British Columbia near the village of Likely (see Figure 1-1). The Spanish Mountain Project consists of about 6,000 ha centred on Spanish Mountain in a single deposit. Access to the area is gained via a paved road commonly referred to as the Likely road, north and east from Williams Lake, BC, leaving Provincial Highway 97 at the village of 150 Mile House, and continuing to the village of Likely, a distance of 85 km.

Figure 1-1: Location Map



The study concludes that the deposit could be developed by conventional open pit mining methods. A site layout for the Spanish Mountain Project has been prepared to illustrate the infrastructure, mining and processing locations for the Project (Figure 1-2).

Figure 1-2: Spanish Mountain Site Layout



Spanish Mountain Gold Ltd. owns the property and has retained AGP Mining Consultants (AGP) to provide a National Instrument (NI) 43-101 compliant Preliminary Economic Assessment (PEA) of the project. As part of the PEA, a new resource estimate was created for use in the study. The resource methodology and results are summarized within this document. This PEA document conforms to the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Mineral Resource and Mineral Reserves definitions referred to in NI 43-101, Standards of Disclosure of Mineral Projects.

The PEA study used the new resource estimate together with a gold price assumption of US$950/oz for the pit design and US$1,100/oz for the financials. An exchange rate of C$1.10: US$1.0 was also applied. All costs, unless otherwise noted, are in Q3 2010 Canadian dollars. Cost estimates were developed for all disciplines both in operating and capital requirements. AGP concludes that the Spanish Mountain Project has the potential to yield a pre-tax NPV of $209 million at a discount rate of 5% with an IRR of 14.7%.

This PEA is preliminary in nature and mineral resource estimates referred to within this study include the use of Inferred resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the preliminary assessment will be realized.

AGP recommends that the Spanish Mountain project advance to the next level of study, pre-feasibility, with geology drilling, engineering, and environmental field programs to support that level of study. Advanced planning of the pre-feasibility study is a critical component to its success and thus AGP has provided recommendations by discipline to ensure sufficient information is provided going forward.

Opportunities exist to reduce both operating and capital costs outlined. Detailed ARD classification of waste material can help planning activities for both the mining and environmental disciplines. Exploration efforts to increase the resource classification and/or contribute more resources to the project will be important in advancing the project.

With the current level of information for the Project, AGP does not see any issues regarding resources, potential economics, or environment, which would inhibit the project from advancing. A decision from Spanish Mountain Gold Ltd. will need to be made to complete the pre-feasibility study including the drilling, metallurgy, geotechnical, and environmental programs to support that level of study.

1.1 Geology

The SMP lies within the Quesnel Terrane, where this has been overthrust from the west onto the pericratonic Kootenay Terrane (Wheeler et al., 1991). At a broad overview scale, Wheeler and McFeely (1991) include the strata underlying the property in the Nicola Group alkaline arc volcanic and associated clastic sedimentary rocks of upper Triassic–lower Jurassic age.

The region has been strongly affected by fold and thrust deformations. Both features are readily identified in outcrop.

The SMP is typically overburdened by up to 10 m of glacial deposits consisting of gravels, sand, till, and local colluvium. Outcrop exposures on the property consist mainly of sedimentary rocks, minor volcanic facies, and minor intrusive rocks.

At present, the Spanish Mountain deposit is classified as an orogenic gold deposit, or a Sediment Hosted Vein (SHV) deposit as defined by Klipfel (2005). By far the most significant gold mineralization at Spanish Mountain is hosted in wide zones (10 m to 135 m) within argillite/siltstone and lesser wacke sequences. Although a minor component, quartz veins with free gold have produced the highest-grade individual samples on the property.

Drilling to date has identified two main styles of gold mineralization. Gold is found within bedded sediments and, to a lesser degree, in quartz veins. Drilling has identified mineralization at Spanish Mountain in an area that extends approximately 1,500 m x 800 m. From the drill hole data, four high-grade zones were observed to be continuous laterally across multiple drill holes and parallel to bedding.

Drilling in the area of the Spanish Mountain deposit dates from 1947, and continues on the property with the current resource estimate based entirely on the results from diamond drilling during the period 2005 to the end of 2009. In general, holes are located on sections spaced at 50 m intervals, with angle holes spaced 50 m to 100 m apart. Early drilling was planned to sample gold in quartz veins. Since 2008, holes were planned to better sample across the stratigraphy as most of the contained metal is not associated with quartz veins but disseminated in the sediments.

AGP used 426 diamond drill holes completed between 2005 and 2009 for the purpose of geological modelling and resource estimation. AGP modelled lithology representing a simplified version of data presented in the drill logs. Modelling was carried out in GEMS™ Version 6.2.3. Four domains were modelled: overburden, upper argillite, greywacke, and lower argillite. AGP also modelled grade shells based on laterally continuous zones where drill hole data was greater than 0.6 g/t Au.

Block grades were interpolated from the drill hole composites. Ordinary Kriging was applied to blocks in one of the high-grade domains and to the blocks outside of the high-grade domains. Inverse Distance weighted to the second power was applied to blocks inside three of the high-grade domains.

The mineral resources at Spanish Mountain were classified by AGP as Measured, Indicated, and Inferred.

1.2 Resource Statement

This PEA incorporates Inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them, and that would enable them to be categorized as mineral reserves. Thus inherent in the study is the risk that the value of these Inferred mineral resources may not be realized.

The mineral resource is defined as the material currently considered as mill feed by this study contained within the final pit design. Other mineralized zones exist beneath the present design for which this study was not able to incorporate with the assumed gold price of US$950/oz. Future studies with more detailed information on costs and a potentially higher gold price assumption will one day be brought into production.

Table 1-1 outlines the resources by classification for a 0.2 g/t cutoff on gold. Note that the grades have been diluted.

Table 1-1: Spanish Mountain Gold Diluted Resource Summary (0.2 g/t Au cutoff)

	Units	Measured	Indicated	Measured + Indicated	Inferred
Resource Tonnage	Tonnes	4,875,900	72,498,800	77,374,700	39,531,300
Diluted Gold Grade	g/t	1.04	0.50	0.53	0.47

1.3 Geotechnical

The preliminary engineering geology of the Spanish Mountain deposit has been summarized to provide a basis for scoping level mine planning and preliminary economic assessments. BGC has developed a basic description of the expected geologic materials of the resource area from available maps, geologic descriptions by Spanish Mountain, corehole data, and field review. The five preliminary geotechnical units for mine design are siltstone, argillite, greywacke, conglomerate, and fault zones. Relatively limited data is available regarding the rock mass strength and the geologic structure in the Spanish Mountain deposit.

Sufficient data has been compiled regarding geotechnical strengths of the primary rock types to provide a range of potential pit wall angles for use in the preliminary economic assessment. However, in order to develop the slope design angles presented in this report, numerous assumptions had to be made about the potential primary controls on slope stability, the geology, the strength of the rock mass, the groundwater pressures, and the potential failure mechanisms. The following assumptions were made:

- Inter-ramp slope angles could be limited due to structurally controlled failures along continuous bedding
- anisotropy of the rock mass was not considered in the generic (i.e., rock mass) stability analyses conducted
- groundwater pressures were assumed to be a function of the lithostatic stress.

Despite considerable scatter in the bedding orientations, both the oriented core and surface mapping data in the Main Zone and North Zones identify two to three prominent bedding orientations. Depending on the local continuity of the bedding and the shear strength of these discontinuities, bench scale failures may occur for pit wall orientations that slope parallel to the bedding dips, and occasional wide berms may be required to contain the failures. Bedding plane structural discontinuities could control achievable inter-ramp angles on north facing slopes in the Main Zone, and for northeast to southwest facing slopes in the North Zone. At this preliminary stage of design, it is recommended that bedding should not be undercut where the average dip is greater than 30°, in order to minimize the potential for structurally controlled instability.

Based on the estimated rock mass strength of the argillites in the Main Zone, overall pit wall angles of 32° to 43° are predicted to be feasible for pit wall heights between 250 m and 500 m. Due to the relatively high rock mass strength of the greywacke, significantly steeper overall slopes could be achieved; however, it appears that the critical south wall will be primarily in the footwall argillites.

Based on the estimated rock mass strength of the siltstone in the North Zone, and assuming “dry” conditions, overall pit wall angles ranging from 42° to 55° could potentially be achieved for slope heights from 100 m to 200 m. However, partially saturated conditions are likely more reasonably assumed due to the presence of Spanish Creek nearby, in which case shallower pit wall angles of between 37° and 48° are predicted. Regardless of the groundwater pressure assumptions in the North Zone, a high degree of depressurization will be required to achieve reasonable slope angles in the siltstone. Groundwater pressures will need to be more accurately quantified in the proposed pit walls before greater confidence can be gained in the design angles for these materials. Piezometers installed in 2010 should provide the necessary information on groundwater pressures in the proposed pit walls to further evaluate the feasibility of the design angles proposed in the PEA.

Engineering geology interpretations and pit wall design angles presented in this report are based on adequate information for scoping level designs, but should be considered preliminary.

1.4 Mining

A single deposit is exploited at Spanish Mountain. The plant throughput is designed for 40,000 t/d with the mining fleet to match. A single open pit is mined in four phases with waste material placed adjacent to the pit, near the plant, backfilled in Phase 2 and a portion of the PAG material hauled to the TMF to be stored subaqueously.

The open pit is developed using conventional rotary drilling, blasting and loading with hydraulic shovels and 180-tonne trucks. The drills will be diesel powered to facilitate movement within the pits, while the hydraulic shovels will be electric powered to reduce operating costs. The open pit mine will have a LOM strip ratio of 1.97:1. A total of 77.4 Mt of Measured and Indicated resource will be supplied to the mill from the open pit with an additional 39.5 Mt of Inferred material. Waste will total 230.1 Mt.

Phase 2 or the North Zone pit will be backfilled when mining is complete in that Phase in Year 7. A total of 45.6 Mt will be backfilled or approximately 20% of the total waste material.

Mill feed material will be mined starting in Year -2 during the pre-stripping of the mine. This will be stored adjacent to the primary crusher location. It will reach a maximum tonnage of 5.6 Mt prior to plant production commencing. Waste during this period will be stored in the TMF footprint; NAG in the embankments and PAG lining the base of the TMF. A small amount of NAG material will be used to build access roads around the pit area and on the site. Mining will commence sufficient to provide the plant 40,000 t/d of feed material in Year 1 and continue at that rate until Year 7 when production will start to taper off as the mining occurs in a single phase. Mining will be completed in Year 10. The stockpiled mill material stored in the pre-stripping will be drawn down by the end of Year 2.

The mining equipment fleet is considered to be fully leased. No capital costs for mining are included in the calculations.

Mine operating costs are estimated at $3.75/t of mill feed with an additional $0.85/t of mill feed for leasing. A financing charge of 3% is assumed for mining equipment leasing and has been included in the operating costs. This equates to US$231/oz for mining and another US$53/oz for leasing for a total of US$284/oz. Life-of-mine costs of $1.26/t of total material are projected for the Spanish Mountain Project.

1.5 Process Design and Metallurgy

1.5.1 Process Plant Design and Metallurgy

Grindability, gravity concentration, flotation, and cyanidation testwork has been carried out on three composite samples from the deposit. The composites had a head gold grade varying from 0.45 to 0.94 g/t Au and represented different lithologies in the deposit (see Table 1-2).

Table 1-2: Average Head Analysis Results for the Metallurgical Composites

Composite	S (%)	C (%)	S_{so4} (%)	C_{org} (%)	C_{inorg} (%)	Au (g/t)	Ag (g/t)	Fe (%)
865-1	1.4	3.27	0.02	0.28	3.03	0.45	1.2	4.81
865-2	2.96	3.22	0.03	1.18	2.04	0.94	1.2	4.12
865-3	1.4	2.3	0.02	0.26	2.05	0.82	0.9	3.32

The testwork conducted to date has demonstrated that at a relatively coarse primary grind of 80% passing 184 µm, a gold recovery to rougher concentrate of 95% is readily achievable. Regrinding the rougher concentrate to 80% -20 µm results in good liberation of the gold particles and this permits a cyanidation stage recovery of 95% and an overall flowsheet gold recovery of 90%. Due to preg-robbing carbon present in the deposit ore and flotation concentrate, the leaching step has been determined to be carbon-in-leach (CIL).

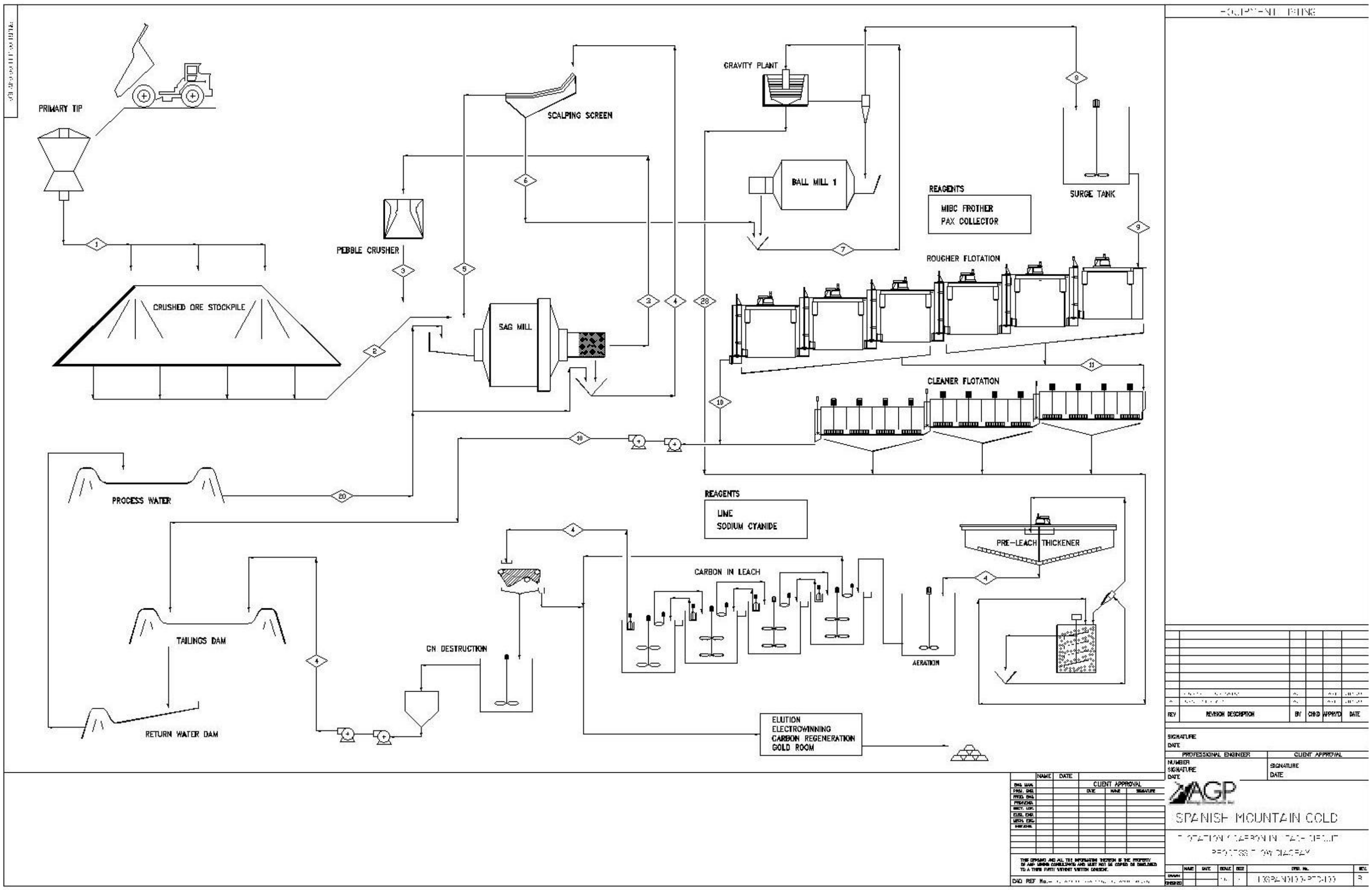
A process design criteria was developed from the metallurgical testwork that provided a platform for a process plant design. The proposed flowsheet is very straightforward and consists of primary crushing followed by SAG milling, closed circuit ball milling with gravity recovery, froth flotation, regrinding of the concentrate, dewatering, CIL cyanidation, elution, and gold electrowinning. This is a standard flotation/cyanidation flowsheet in use elsewhere in the world and poses low technical risk. A process plant flowsheet is shown in Figure 1-3.

In addition to a conceptual process flowsheet, preliminary plant layout drawings (plans and elevations) were prepared and used to develop operating and capital cost estimates for the process plant.

For the 40,000 t/d processing option, a direct capital cost of $215 million is estimated to ±40% level of accuracy.

Process Plant operating costs of $5.12/t are estimated (including tailings) with consumables (reagents, grinding media and wear items) accounting for approximately 60% of this cost.

Figure 1-3: Spanish Mountain Process Plant Flowsheet



1.5.2 Tailings Management Facility

The principal objective of the tailings management facility (TMF) is to provide secure containment of all tailings solids and a portion of the potentially acid generating (PAG) waste rock.

Ore processing will produce two tailings streams, rougher scavenger tailings (RST) and cleaner scavenger tailings (CST), which will be transported from the plant site to the TMF in separate pipelines at an average solids content of 30% by weight. Each tailings stream will be deposited independently; the RST will be discharged along the TMF embankments to create tailings beaches, and the CST will be discharged to allow for progressive encapsulation by the RST and saturation by the supernatant pond. The PAG waste rock will also be deposited to allow for progressive encapsulation by the RST and saturation by the supernatant pond.

The starter TMF will be constructed during the pre-production phase and is sized to store the estimated volume of tailings and PAG waste rock produced during the first two years of operation, plus the supernatant pond volume and allowances for wave run-up, post-seismic settlement, sloping beaches, and containment of the inflow design flood. The TMF embankments will be constructed in annual stages with each stage providing the required capacity for the period until the next stage is completed, with a final storage capacity of approximately 126 Mt of tailings, 14 Mt of PAG waste rock, plus the supernatant pond volume and freeboard allowances.

1.6 Infrastructure and Site Layout

The mill will be constructed to the west of the open pit and consists of the processing plant and the supporting infrastructure for the mining operations. Access to the site will be on the existing forest access/exploration road 6 km from the Town of Likely and will require upgrades. The anticipated power demand for the entire mine site is approximately 34 MW. For the basis of the study, the power line feeding the site will consist of an upgraded portion (from Soda Creek area to Gavin) and a newly constructed circuit from Gavin to site. A mining equipment garage as well as a dry, offices, and warehouse is included in the site complex.

Refer to Figure 1-2 for the overall site plan.

1.7 Capital and Operating Costs

1.7.1 Capital Costs

The capital costs for the Spanish Mountain Gold project are summarized in Table 1-3. The costs are based on the estimate for a 40,000 t/d processing plant using a standard floatation with Carbon in Leach circuit and gold electrowinning. The mine has a 10-year life with full production at 40,000 t/d for the first six years then tapering off until the mine is complete.

Table 1-3: Spanish Mountain Capital Cost Summary

Capital Category	Total Capital (M$)	Pre-Production Capital Year – 2 to Year -1 (M$)	Production Capital Year 1 (M$)	Sustaining Capital Year 2+ (M$)
Open Pit Mining	-	-	-	-
Processing	215.0	170.3	42.6	2.1
Infrastructure	87.4	77.1	1.2	9.1
Environmental	18.5	18.5	-	-
Indirects	70.4	57.4	9.5	3.5
Contingency	72.1	58.9	11.0	2.2
Total	**463.4**	**382.2**	**64.3**	**16.9**

Initial capital requirements (Pre-production) as shown are $382.2 million. It should be noted that the open pit equipment has been considered under a full lease for this study. Production starts in Year 1 and the capital requirements may be partially offset by revenue in that year. Capital requirements for Year 1 total $64.3 million. The indirect and contingency values varied by capital cost item. The indirect and contingency values referred to in Table 1-4 are percentages of the direct capital numbers. These percentages are calculated from various areas within each capital category, reason why the percentages may not be an even number.

Table 1-4: Indirect and Contingency Percentages by Capital Category

Capital Category	Indirects (%)	Contingency (%)
Open Pit Mining	10.0	15.0
Processing	21.5	25.2
Infrastructure	27.6	18.1
Environmental	0.0	20.0

1.7.2 Operating Costs

Operating cost development is for a 40,000 t/d mining and milling operation running for 10 years. This production rate was chosen in a trade-off study because it offered improved economics over lower production rates with anticipated higher gold prices and the known resource. A single open pit is mined in four phases with waste material placed adjacent to the pit, near the plant, backfilled in Phase 2, and PAG material hauled to the TMF and stored subaqueously. A total of 45.6 Mt will be backfilled or approximately 20% of the total waste material.

All prices in this PEA are quoted in 3Q 2010 Canadian dollars unless otherwise noted. Where an exchange rate to United States dollars is applied, a rate of C$1.10:US$1.00 is considered. Diesel fuel is assumed to cost $0.73/L and electricity costs $0.04/kWh.

The open pit is developed using conventional rotary drilling, blasting and loading with hydraulic shovels and 180-tonne trucks. The drills will be diesel powered to facilitate movement within the pits, while the hydraulic shovels will be electric powered to reduce operating costs. The open pit mine will have a LOM strip ratio of 1.97:1. A total of 77.4 Mt of Measured and Indicated resource will be supplied to the mill from the open pit. Inferred material to be supplied to the mill will amount to 39.5 Mt. Total waste movement will equal 230.1 Mt. Mining will be sufficient to provide the mill with 40,000 t/d from Year 1 with a ramp up period and continue at that rate until Year 7 when production will start to taper off as the mining occurs in a single phase. Mining will be completed in Year 10.

The process plant is designed to operate at a nominal tonnage of 40,000 t/d with feed material from the mine. The first two years will deplete the stockpile created during the pre-stripping. The plant will use conventional grinding and flotation, with a CIL circuit and electrowinning to make a gold doré. Tailings will drain by gravity downhill to the TMF a distance of 2.7 km from the plant.

General and Administrative costs are based on 13 salaried staff and 31 hourly personnel. Employees will be located in the immediate area and no camp is planned or required.

The mining fleet was assumed to be fully leased for this study.

Table 1-5 shows a summary of all operating cost categories on a cost per tonne mill feed basis and a cost per recovered gold ounce.

Table 1-5: Spanish Mountain Project Operating Costs

Cost Centre	Total Operating Cost (M$)	Cost Per Tonne ($/t mill feed)	Cost per Ounce ($US/oz)
Open Pit – Mill Feed and Waste	437.9	3.75	231
Leasing Cost	99.6	0.85	53
Processing	598.1	5.12	315
G&A	49.9	0.43	26
Total	**1,185.4**	**10.15**	**625**

1.8 Economic Analysis

The completion of a trade-off study indicated that with higher gold prices, greater value could be obtained from a production rate of 40,000 t/d day of plant feed. This was advanced in this study and is the chosen case for the project with the Financial Base Case gold price of US$1,100/oz. The pit design was developed using a gold price of US$950/oz.

All prices quoted are in Q3 Canadian dollars unless otherwise noted. An exchange rate of C$1.10 to US$1.00 was used.

In the development of the operating costs for the DCF, the impact of leasing was considered. The potential economics improved as a result of its inclusion and were adopted as the chosen case. The results of the DCF for the 40,000 t/d case with and without leasing have been shown in Table 1-6.

The financial base case gold price is US$1,100/oz. Payables for gold were 99.5%. No royalty was applied for the calculation of NPV in the cash flow.

The results of the DCF for the 40,000 t/d with leasing case indicated that the project has a pre-tax NPV of $209 million at a discount rate of 5% with an IRR of 14.7%. This is an improvement of $16 million in the pre-tax NPV over the non-leasing casing, which had an IRR of only 13.2%. Payback on the project from the start of commercial production is 4.1 years.

Table 1-6: Discounted Cash Flow Results

Cost Category	Unit	40,000 (t/d)	Leasing 40,000 (t/d)
Operating Cost			
Open Pit Mining	($M)	438	438
Lease Cost	($M)	-	99
Processing	($M)	598	598
G&A	($M)	50	50
Sub-total Operating Costs	**($M)**	**1,086**	**1,185**
Capital Costs			
Open Pit Mining	($M)	86	-
Processing	($M)	215	215
Infrastructure – Site	($M)	51	51
Infrastructure – Tailings	($M)	37	37
Environment Costs	($M)	18	18
Indirect	($M)	79	70
Contingency	($M)	85	72
Sub-total Capital	($M)	571	463
Revenue (after refining, payables)	($M)	2,060	2,060
Net Present Value (NPV)			
NPV @ 0%	($M)	404	411
NPV @ 5%	($M)	193	209
NPV @ 8%	($M)	106	125
IRR	(%)	13.2	14.7
Payback Period	**Years (Year paid)**	**4.3 (Year 5)**	**4.1 (Year 5)**

Table 1-7: Metal Production Statistics, Cash Cost Calculation, and Key Economic Parameters

Item	Indicator	Units	Value
Gold	Average Annual Production	oz	172,400
	Initial 5 Year Average Annual Production	oz	213,800
	Total LOM Production	Moz	1.72
Cash Cost	Average Life of Mine Gold Cash Cost	US$/oz	625
	Initial 5 Year Average Gold Cash Cost	US$/oz	570
Key Parameters	Operating Cost	$/t plant feed	10.14
	Mine Life	years	10
	Average Plant Feed Grade	g/t	0.51
	Overall Gold Recovery	%	90
	Initial Capital Cost	$M	447
	Total Capital Cost	$M	463

The project sensitivity to various inputs was examined on the 40,000 t/d case with leasing. The items that were varied were:

- gold price
- gold recovery
- capital costs
- operating cost.

The results of that analysis have been shown in Table 1-8 and Table 1-9. Note that the recovery was held to 100%.

What this indicated was that the project is most sensitive to gold recovery and metal prices. It is the least sensitive to the capital costs. With the mining equipment fully leased, this has further helped to reduce the impact of capital on the overall project. Metallurgical recovery is expected to be stable over a wide range of feed grades. Therefore, while a sensitivity exists, actual practice may show less fluctuation than considered in this analysis.

Table 1-8: Sensitivity Analysis – NPV at 5% Discount Rate

Sensitivity	Unit	Recovery	Metal Prices	Capital Cost	Operating Cost
-20%	($M)	-90	-92	309	382
-10%	($M)	59	58	259	296
Financial Base	**($M)**	**209**	**209**	**209**	**209**
+10%	($M)	359	360	159	123
+20%	($M)	375	511	109	36

Table 1-9: Sensitivity Analysis – IRR

Sensitivity	Unit	Recovery	Metal Prices	Capital Cost	Operating Cost
-20%	(%)	0.0	-0.2	21.6	21.7
-10%	(%)	8.0	7.9	17.9	18.3
Financial Base	**(%)**	**14.7**	**14.7**	**14.7**	**14.7**
+10%	(%)	20.6	20.7	11.9	10.9
+20%	(%)	21.3	26.1	9.5	6.8

The greatest sensitivity in the project is to gold price. With the current financial base price of US$1,100/oz, this is still $303/oz less than spot price of US$1,403 as of 13 December 2010. The sensitivity of the current project to metal prices was also examined. The pit design and schedule did not change, only the value of the gold. The results have been shown in Table 1-10.

Table 1-10: Sensitivity Analysis – Gold Price Impact

Gold Price (US$/oz)	NPV ($M) @ 0%	NPV ($M) @ 5%	NPV ($M) @ 8%	IRR%
950	128	4	-47	5.2
1,000	222	72	10	8.6
1,050	317	141	68	11.7
1,100 (Financial Base)	**411**	**209**	**125**	**14.7**
1,150	505	278	182	17.5
1,200	600	346	240	20.2
1,250	694	415	297	22.7
1,300	788	483	355	25.2

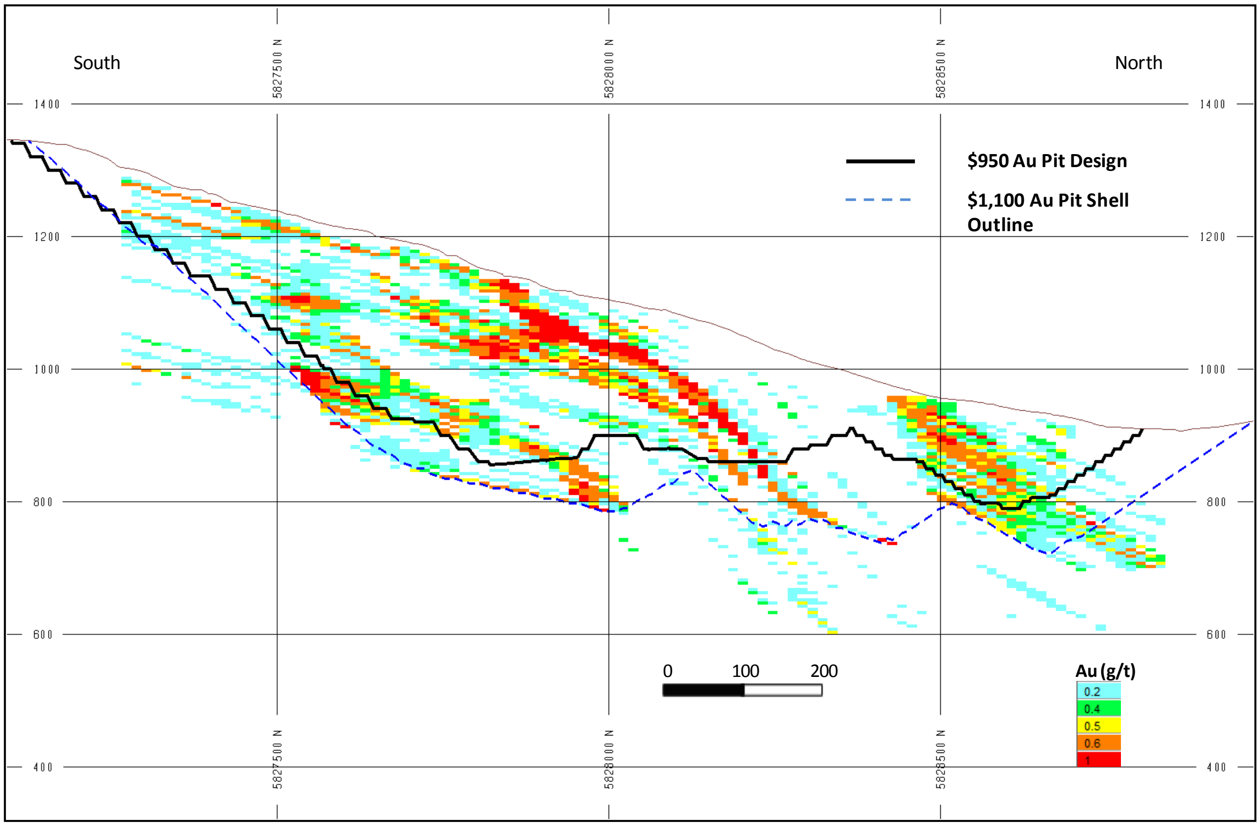
The open pit design was based on the use of a gold price of US$950/oz to allow for potential price fluctuations. To determine the impact of a higher price on open pit potential, a series of pit shells were developed for gold prices in excess of US$950/oz. Using the initial parameters from the trade-off study for the 40,000 t/d case, four additional pits were developed. These were compared to the original US$950 pit shell in Table 1-11.

What this analysis indicated was that with a gold price of US$1,100/oz, the resulting shell could potentially provide 29% more material suitable for plant feed with a grade of 0.44 g/t. Contained ounces of gold within this shell could potentially increase from 2.5 Moz to 2.9 Moz. It should be noted that the pit shell ounces are not the exact ounces that would be extracted in the final pit design but provides a good indication of the potential that may exist with additional drilling and detailed design work. The US$1,100/oz pit shell has been shown in cross section with the existing pit design in Figure 1-4. The pit shell is indicated by the dashed line.

Table 1-11: Sensitivity Analysis – Pit Shell Size to Gold Price

Item	Units	US$950/oz	US$975/oz	US$1,000 /oz	US$1,050 /oz	US$1,100 /oz
Mining Cutoff	g/t	0.24	0.23	0.23	0.22	0.21
Milling Cutoff	g/t	0.19	0.18	0.18	0.17	0.16
Plant Feed	Mt	158.6	170.5	175.8	189.4	203.8
Plant Feed	g/t	0.49	0.48	0.48	0.46	0.44
Waste	Mt	278.3	301.9	304.5	310.2	321.3
Total Material	Mt	436.9	472.4	480.3	499.6	525.1
Strip Ratio		1.76	1.77	1.73	1.64	1.58
In Situ Gold	Moz	2.52	2.65	2.69	2.80	2.91

Figure 1-4: Pit Design vs.US$1,100/oz Pit Shell



1.9 Environmental

The Spanish Mountain Project includes mine development components located within the Cedar Creek and Spanish Creek watersheds. The TMF is located in the Cedar Creek watershed and the deposit, waste dumps, and the plant site are located in the Spanish Creek watershed.

Project-specific environmental studies have been conducted since 2007, including aquatic resource studies (water quality and quantity, sediment quality), aquatic biota studies (fish species and community composition, fish habitat, primary, and secondary productivity), terrestrial resource studies (wildlife and vegetation) and climatology.

Water samples taken within the claim boundary have consistently shown concentrations of total and dissolved metals exceeding provincial and federal guidelines for the protection of aquatic life, likely due to natural mineralogy of the claim area and disturbance from historic placer mining activities. Samples collected outside of the claim boundary are generally within provincial and federal guidelines.

Rainbow trout have been captured in the Cedar Creek, Spanish Creek, and Winkley Creek systems during the baseline sampling programs. In addition, Chinook salmon, dace, and burbot were captured in the Cedar Creek system, Chinook juveniles were captured, and Coho salmon adults were detected near the mouth of Spanish Creek. Historical records indicate that sockeye salmon and bull trout are also present in the Spanish Creek watershed; however, a series of falls and rapids in the lower reaches of Spanish Creek obstruct the upstream movement of anadromous fish.

Grizzly bears, black bears, caribou, bighorn sheep, moose, fishers, and wolverines are common in the biogeoclimatic zones of the Spanish Mountain Project, the Interior Cedar–Hemlock (ICH) and Engelmann Spruce-Subalpine Fir (ESSF) biogeoclimatic zones. The Project is adjacent to the range of the Wells Grey herd of mountain caribou, a threatened species with a declining population trend.

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. A typical EA is generally completed within a two-three year period. The federal *Fisheries Act* prohibits the harmful alteration, disruption, or destruction of fish habitat without specific authorization. Construction of the TSF in the Nina Lake basin of the Cedar Creek watershed will 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 EAO Project Information Centre (e-PIC) and federal CEA registry. The project will be in the traditional territories of the T'exelc (Williams Lake) and Xats'ull Indian Bands.

2 INTRODUCTION AND TERMS OF REFERENCE

This report describes the results of the Preliminary Economic Assessment (PEA) study for the Spanish Mountain Project, currently owned by Spanish Mountain Gold Ltd. (SMG). This engineering and financial analysis was done using NI 43-101 compliant resources within an open pit design. All monetary amounts are provided in 3Q 2010 Canadian dollars unless otherwise noted. All units used in this report are metric; grid references are based on the UTM NAD 83 coordinate system unless otherwise stated.

This report is prepared in accordance with disclosure and reporting requirements set forth in National Instrument 43-101 (NI 43-101), Companion Policy 43-101CP, and Form 43-101F1, and complies with Canadian National Instrument 43-101 for the 'Standards of Disclosure for Mineral Projects' for the Canadian Securities Administration. It was prepared at the request of Mr. Ron Halas, Chief Operating Officer of SMG. The following people are listed as leads in each discipline on the PEA project team. Qualified Persons (QP) as set out in NI 43-101 are also designated by a "QP" and are listed as such in the Certificates of Qualified Persons section. QPs who visited the property from the 20th to the 21st of April 2010, and from the 3rd to the 5th of August 2010, are properly identified.

Michael Waldegger, P.Geo.	–	Geologist, AGP (*QP, Site Visit – August 2010*)
Gordon Zurowski, P.Eng.	–	Mining (Open Pit/Financials), AGP *(QP, Site Visit – April 2010)*
Warren Newcomen, P.Eng.	–	Geotechnical, BGC *(QP, Site Visit – April 2010)*
Mario Colantonio, P.Eng.	–	Infrastructure, PES *(QP, Site Visit – April 2010)*
Andy Holloway, P.Eng.	–	Processing/Metallurgy, AGP *(QP)*
Morris Beattie, P.Eng.	–	Metallurgy, Beattie Consulting Ltd. *(QP, Site Visit – October 2009 and September 2010)*
Ken Brouwer, P.Eng	–	Environmental, Knight Piésold *(QP)*.

During the site visit, items such as transportation routes, mining site, layout of infrastructure, geology, geotechnical assessment of the drill core, and mine staffing issues were assessed. This satisfies the condition of a site visit performed by an independent qualified person for NI 43-101 regulations.

Information, conclusions, and recommendations contained herein are based on a field examination, including a study of relevant and available technical data including and not limited to the numerous reports listed in the References section. Valuable site-specific information was provided by Ron Halas and also Stuart Morris, SMG Vice-President, Development Geology.

Much of the information contained in this report from Section 4.0 to Section 15.0 was extracted from a report from Spanish Mountain Gold, titled "Updated Resource Estimation Report on the Spanish Mountain Gold Deposit, 1 May 2009." The current AGP Preliminary Economic Assessment report is updated with the most recent information available at the time of study. All reports are available on SEDAR for public release.

3 RELIANCE ON OTHER EXPERTS

AGP has followed standard professional procedures in preparing the content of this report. Data used in this report has been verified where possible and this report is based upon information believed to be accurate at the time of completion. AGP has no reason to believe that the data was not collected in a professional manner. AGP has not verified the legal status or legal title to any claims and the legality of any underlying agreements that may exist concerning the Property.

The authors have also relied on several sources of information on the property, including digital geological and assay data. Therefore, in writing this report, the qualified persons rely on the truth and accuracy as presented in various sources listed in the references section of this report. Qualified persons as defined by the NI 43-101 regulations are relied upon for each section of this report. The sections that the qualified persons are responsible for are indicated in Section 30, Certificates of Qualified Persons.

Critical areas of information that are relied upon in this report are the geological resource estimation and metallurgy. The NI 43-101 compliant resource report for Spanish Mountain has been prepared by AGP. Michael Waldegger, P.Geo. of AGP created this resource estimate for use in the current PEA study. G&T Metallurgical has conducted metallurgical testwork from 2007 to 2010 under the initial supervision of Gary Hawthorn, P.Eng. (2007) and more recently by Morris Beattie, P.Eng. of Beattie Consulting Ltd. The testwork is referenced in Section 16 of this PEA report. Andy Holloway, P.Eng., has confirmed the validity of the testwork used in the PEA study with AGP.

AGP Mining Consultants Inc. (AGP) based in Barrie, Ontario, Canada managed the project. AGP collaborated with BGC Engineering Inc. (BGC) based in Vancouver, British Columbia, Canada for Geotechnical, Porcupine Engineering Services (PES) based in Timmins, Ontario, Canada for Infrastructure, and Dowding Reynard & Associates (DRA) based in Peterborough, Ontario, for Processing.

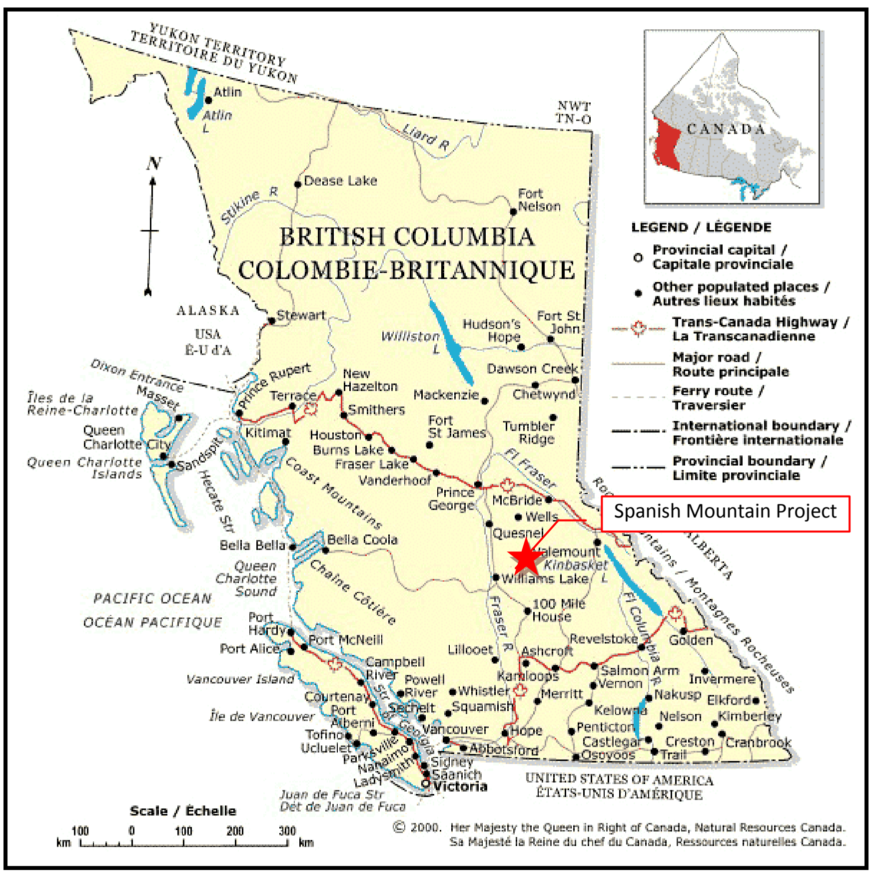
The Tailings Management Facility discussion – Section 18, Environmental – Section 22, and the Project Implementation Plan (pertaining to permitting) – Section 26, of the report were done in collaboration with Alexis McPherson and Ken Brouwer of Knight Piésold (KP). The Project Implementation plan was also completed in coordination with Ron Halas of Spanish Mountain Gold.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Spanish Mountain property is located in the Cariboo Mining Division within the Cariboo region of central British Columbia (Figure 4-1). The property is located approximately 70 km northeast of Williams Lake, BC, with the closest population centre being the village of Likely, located about 6 km northwest of the principal area of interest. The Spanish Mountain gold deposit lies just west of the northwest end of Spanish Lake. The centre of the approximately 9 km long x 5 km wide property is at approximate UTM[1] coordinates 604500 East and 5826000 North. The deposit is centred at 604400 East and 5827800 North.

Figure 4-1: Location Map



---

1 Universal Transverse Mercator Grid metric coordinates are NAD (North American Datum) 83, Zone 10.

4.2 Description

The Spanish Mountain property consists of 33 mineral claims (Figure 4-2) and eight overlapping placer claims (Figure 4-3) covering about 6,000 ha centred on Spanish Mountain. All but two of the mineral claims make up a single contiguous block. The property includes no surface rights, with the exception of one small, alienated parcel (DL12083) at the northwest end of Spanish Lake (Figure 4-4).

The claim block is surrounded by mineral claims held by various third parties, and with the exception of the eight placer-claims mentioned above, is underlain by placer claims held by third parties.

A complete list of claims[2], with tenure numbers and type, names where applicable, ownership, and expiry dates is contained in Table 4-1.

Summary details of the underlying agreements in respect of some of the mineral claims, referred to in Table 4-1, are discussed in Section 4.3.F

The presently defined mineral resource is located largely within the boundaries of the CPW claim (No. 204667), but extends southward onto the PESO claim (No. 204021), and claim No. 512542.

[2] Note that this report is concerned principally with the resource estimate for several mineralized zones, covering only a small portion of the overall claim area.

Figure 4-2: Mineral Claims Map

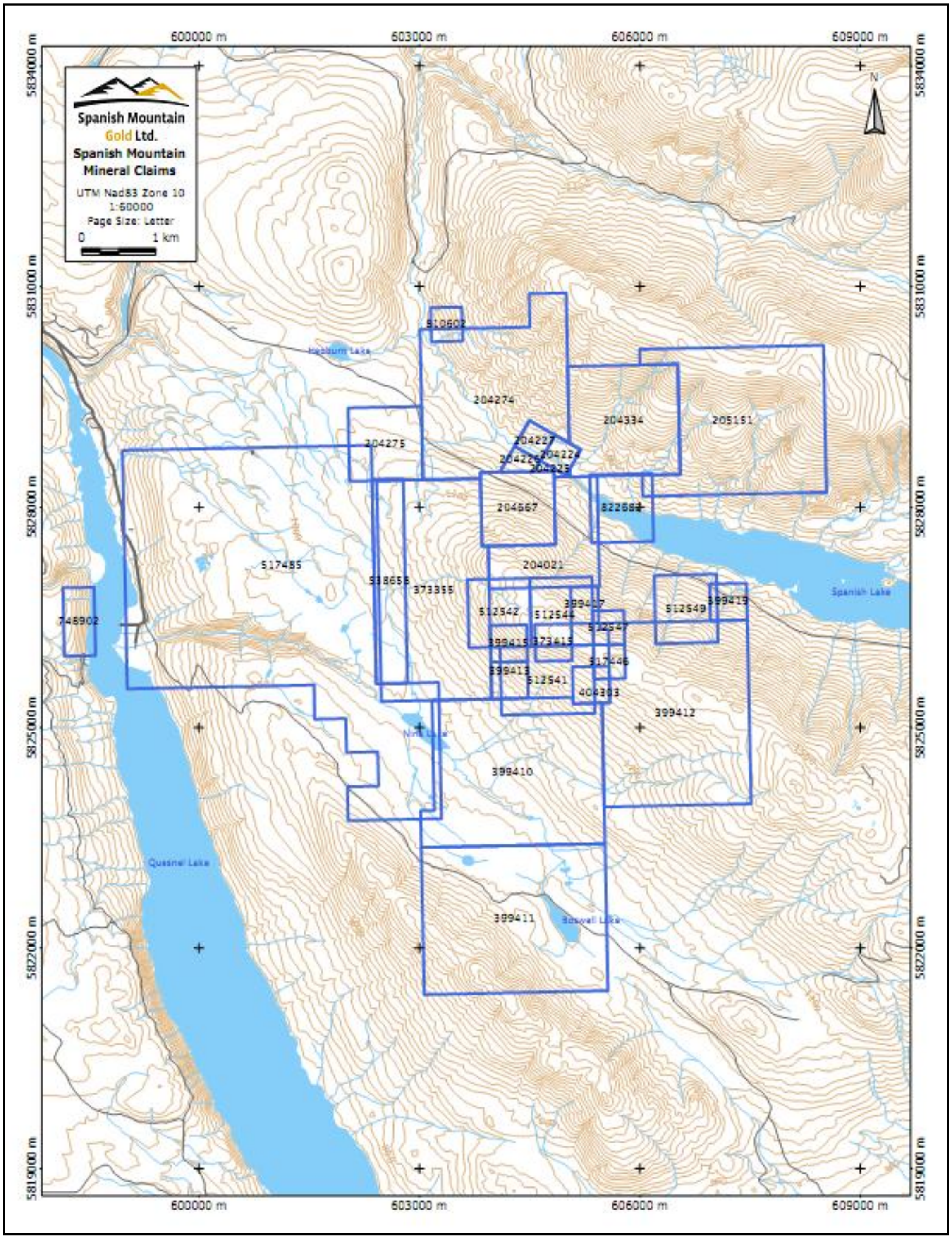


Figure 4-3: Placer Claims Map

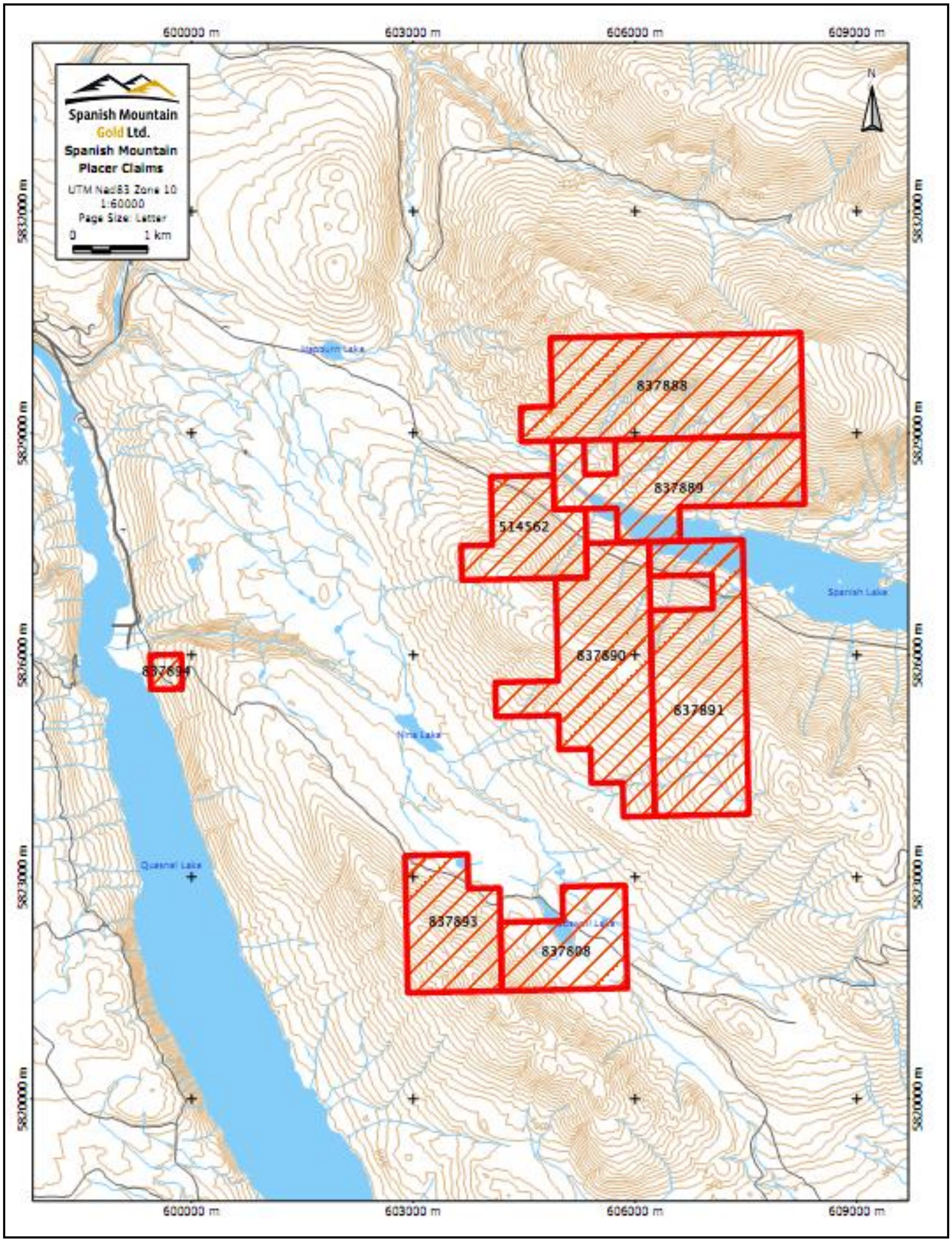


Figure 4-4: Surface Rights Maps

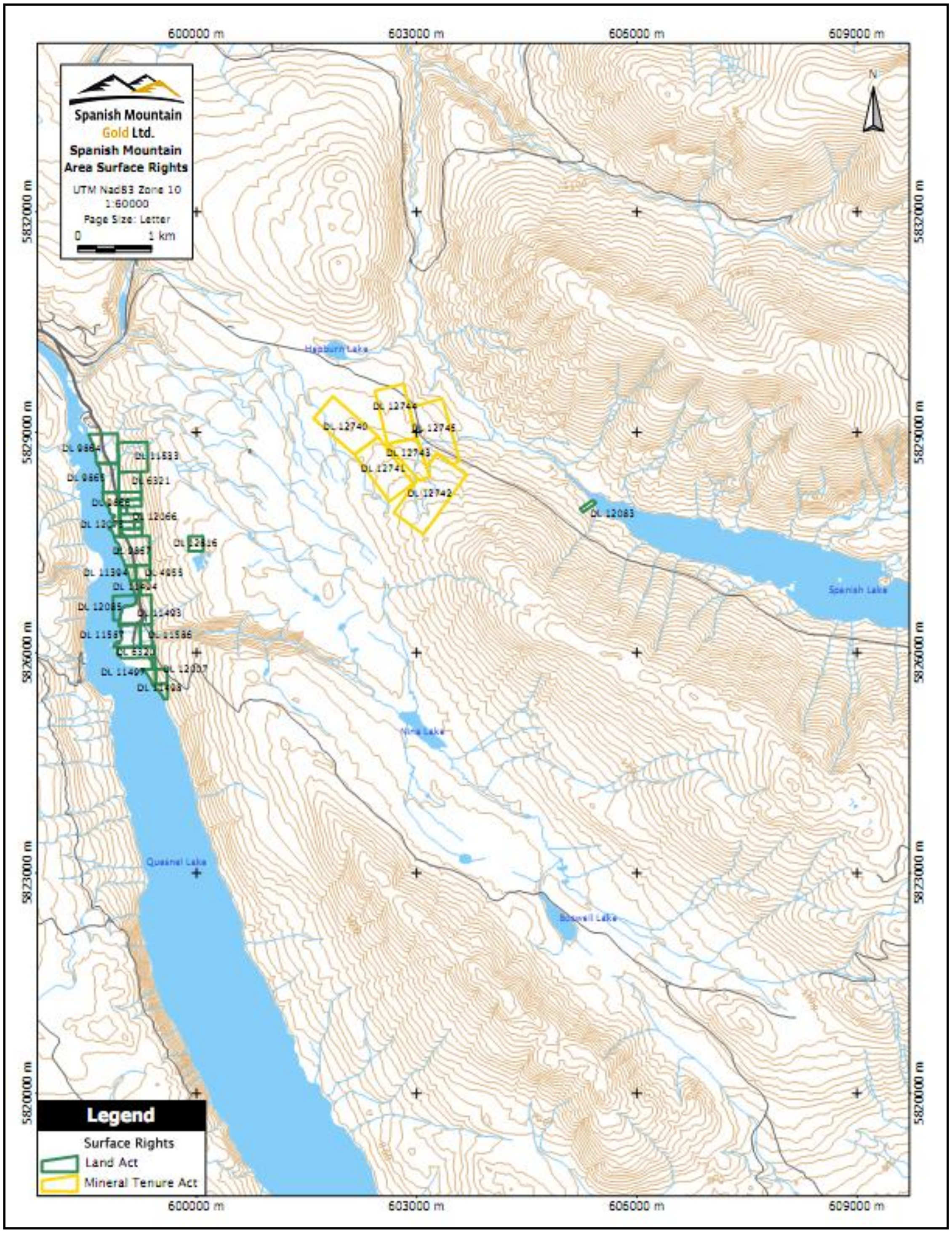


Table 4-1: Summary of Mineral Claims Data

Tenure Number	Tenure Type	Claim Name	Map Number	Owner	Expiry Date
204021**	Mineral	PESO	093A053	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
204224**	Mineral	DON 1	093A053	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
204225**	Mineral	DON 2	093A053	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
204226**	Mineral	DON 3	093A053	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
204227**	Mineral	DON 4	093A053	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
204274**	Mineral	MARCH 1	093A053/063	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
204275**	Mineral	MARCH 2	093A053/063	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
204334**	Mineral	JUL 2	093A053/063	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
204667*	Mineral	CPW	093A053	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
204727**	Mineral	MY 1	093A053	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
205151**	Mineral	MEY 1	093A053/063	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
373355	Mineral	ARMADA	093A053	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
373415**	Mineral	N.R.1	093A053	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
399410	Mineral	ARMADA 2	093A053	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
399411	Mineral	ARMADA 4	093A053	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
399412	Mineral	ARMADA 5	093A053	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
399413	Mineral	ARMADA 6	093A053	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
399415	Mineral	ARMADA 8	093A053	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
399417	Mineral	ARMADA 10	093A053	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
399419	Mineral	ARMADA 12	093A053	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
404303	Mineral	AG 2	093A053	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
512541	Mineral		093A053	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
512542	Mineral		093A053	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
512544**	Mineral		093A053	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
512547	Mineral		093A053	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
512549	Mineral		093A053	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
517446	Mineral		093A053	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01
517485	Mineral		093A053	SPANISH MOUNTAIN GOLD LTD.	2018/jul/28
538658	Mineral	MOORHEAD 14	093A053	SPANISH MOUNTAIN GOLD LTD.	2018/aug/04
748902	Mineral	SPAN 1	093A053	SPANISH MOUNTAIN GOLD LTD.	2011/apr/15
810582	Mineral	SPAN 2	093A044	SPANISH MOUNTAIN GOLD LTD.	2011/jul/07
810602	Mineral	SPAN 3	093A063	SPANISH MOUNTAIN GOLD LTD.	2011/jul/07
822682	Mineral		093A053	SPANISH MOUNTAIN GOLD LTD.	2018/nov/01

Notes: (*) Claim (CPW) highlighted in **blue** subject to the Wallster and McMillan option agreement
(**) Claims highlighted **red** subject to the Mickle option agreement
Claim 512544 is a redefinition of NR2 (373416) in the Mickle option agreement.

4.3 Ownership

Spanish Mountain Gold Inc, with offices at 920-1055 West Hastings St., Vancouver, BC, owns all thirty-three mineral claims comprising the Spanish Mountain property, subject in the case of some of the claims to the terms of underlying option agreements. Spanish Mountain Gold also holds the eight placer claims.

Spanish Mountain Gold Inc. was formerly called Skygold Ventures Ltd. The change in name was effective 14 January 2010.

As shown on Table 4-1, there are two underlying option agreements in respect of certain of the claims in the property:

1. An option agreement dated 10 January 2003 between Wildrose Resources Ltd. (Wildrose) and Robert E. Mickle of Likely, BC, for Wildrose to earn a 100% interest in 12 mineral claims as listed in Table 4-1. The agreement provides for escalating cash payments totalling C$100,000 over five years. These payments have all been made. There is provision for a 2.5% net smelter return (NSR) royalty payable to Mickle for any production from these claims, of which 1% and 1.5% may be purchased by payment of C$500,000 to Mickle. There is also a requirement to expend in aggregate at least C$200,000 on the Mickle claims during years six to ten of the agreement.
2. An option agreement dated 20 January 2003 between Wildrose (the Optionee), Spanish Mountain Gold (the Assignee), and D E. Wallster as to a two-thirds interest and J.P. 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 CPW mineral claim. The agreement provides for escalating cash and/or shares of equal value payments totalling C$348,000 over nine years, in addition to 30,000 common shares of the Assignee on signing. There is provision for a 2.5% NSR royalty payable to the Underlyers for any production from the CPW claim, of which 1% may be purchased by payment of C$500,000 to the Underlyers at the commencement of commercial production from the CPW claim.

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

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

Under the proposed Transaction, Wildrose shareholders would receive 0.82 common shares of Spanish Mountain Gold for each common share of Wildrose. Spanish Mountain Gold would assume outstanding warrants and stock options of Wildrose on the basis that each warrant or option of Wildrose will 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 09 July 2008, Spanish Mountain Gold announced that "...all the conditions to the acquisition by Spanish Mountain Gold of Wildrose Resources Ltd. (Wildrose), 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, Spanish Mountain Gold became responsible for the underlying agreements. Further to this, by virtue of the name change in 2010, Spanish Mountain Gold is now responsible for the underlying agreements.

4.4 Taxes and Assessment Work Requirements

All mineral claims in the property are in good standing until various dates as shown on Table 4-1, the earliest date being 15 April 2012.

There are no taxes payables with respect to the property.

4.5 Permits and Liabilities

The British Columbia Government, to cover the estimated cost of reclamation on the property, holds in trust reclamation bonds for the property totalling $65,000. As the project is ongoing, the bonds remain outstanding. There is also in force a "Free Use Permit," No. 19545-60/L47624, which allows for tree removal and for road and drill site building, and is good to 31 December 2012. To the best of our knowledge, there are no outstanding environmental issues that would likely delay or adversely affect the project.

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

5.1 Accessibility

Access to the area is gained via a paved road (the “Gold Rush Trail”), commonly referred to as the Likely road, north and east from Williams Lake, BC, leaving Provincial Highway 97 at the village of 150 Mile House, and continuing to the village of Likely, a distance of 85 km (Figure 5-1). From Likely road, property access is by good secondary roads to the main area of interest by the Cariboo Lake Road and Forest Service Road (FSR 1300), and to the southern portions of the property by Cedar Creek Road (FSR 3900). There is an interconnecting network of logging roads, some maintained and winterized by the company, which provides good access to various work areas.

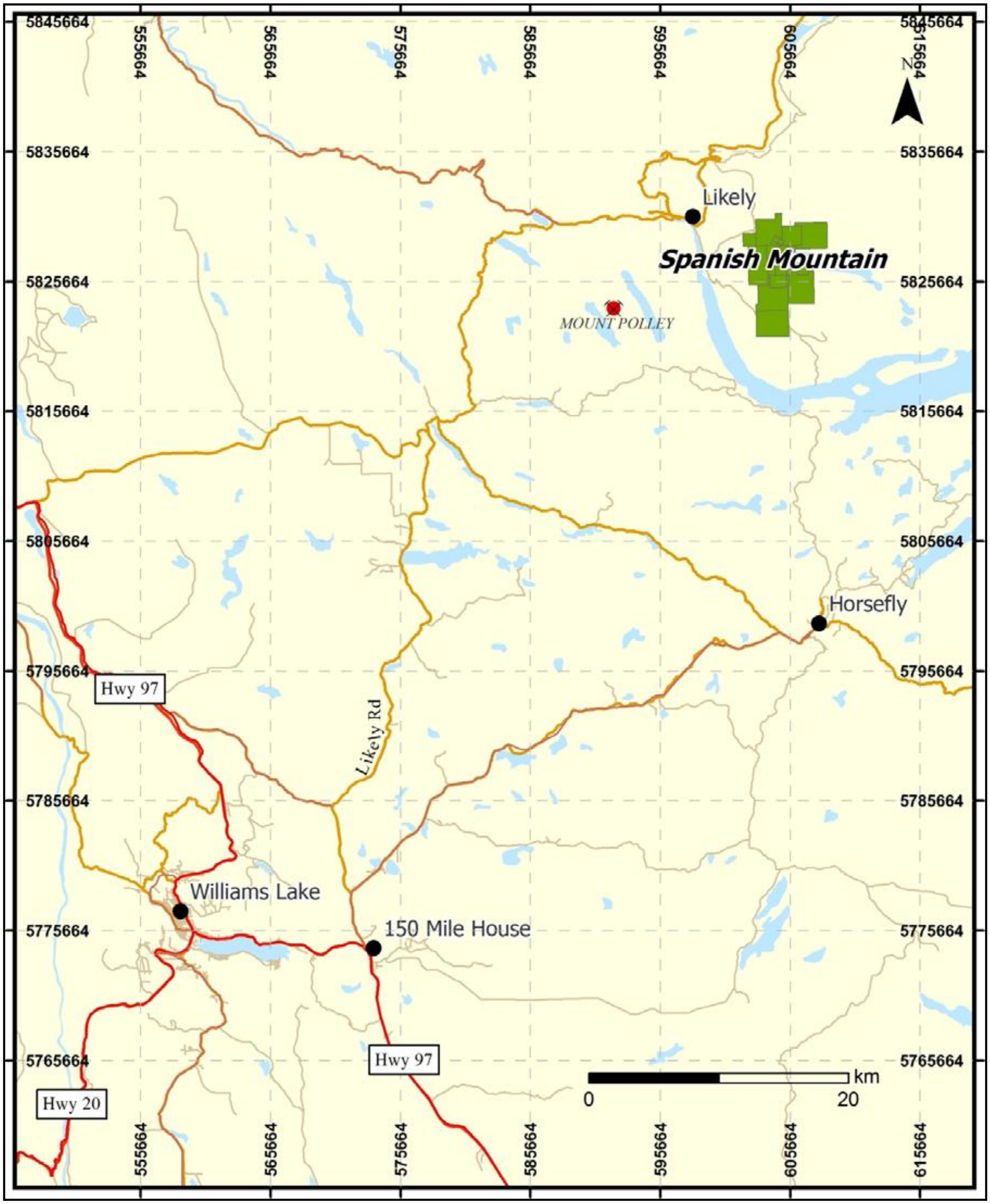
5.2 Climate and Hydrogeology

Two automated weather stations (SMLOW and SMHIGH) and two hydrology-gauging stations (SCSG-10 and SCSG-05) were installed at the Spanish Mountain property in May 2007. The weather stations were installed in the deposit area at elevations of 1,080 masl (SMLOW) and 1,260 masl (SMHIGH). In late August/early September 2010, a new weather station was installed at the Spanish Mountain Camp (SPANISHCAMP) and SMHIGH was relocated to the Cedar Creek catchment (CEDAR).

The two hydrology gauging stations installed in 2007 are located on Spanish Creek; SCSG-10 is located immediately downstream of the outlet of Spanish Creek, and SCSG-05 is located 880 m downstream of SCSG-10. In August 2010, a hydrology gauging station was installed at the outlet of Nina Lake (NINA) and SCSG-05 was removed, and in September 2010, hydrology-gauging stations were installed on Blackbear Creek (BLAC), Cedar Creek upstream of Boswell Creek (CEDA), and near the mouth of Winkley Creek (WINK).

The most recent estimate of mean annual precipitation was presented in KP’s “2007 Environmental Baseline Summary Report” (April 2008) which was prepared when only seven months of site-specific data had been collected. The mean annual precipitation was consequently estimated to be 1,014 mm at an elevation of 1,260 m based on long-term precipitation data at Barkerville, BC. Additional analysis of precipitation conditions using the site-specific data is scheduled to be completed in 2011.

Figure 5-1: Property Access Map



The average annual evaporation on site was estimated using evaporation information from the Mt. Polley Mine (located approximately 15 km east of Spanish Mountain) and adjusting the evaporation values based on elevation. This was computed using an empirical approach suggested by the BC Ministry of the Environment's Manual of Operational Hydrology in BC (2007), which includes a 10% decrease in evaporation with a 350 m increase in elevation. The average annual site evaporation was estimated to be 389 mm at an elevation of 1,260 m.

The climate in the area is modified continental with moderately warm summers and cold snowy winters. Typical daytime temperature ranges are from 25°C to 35°C in summer and -15°C to -35°C in winter. The area lies close to the east of the interior dry belt; precipitation averages about 700 mm (rainfall equivalent) at Likely. Thick accumulations of snow (as much as 2 m) are common in winter on the property.

Drilling programs can be performed on a year-round basis, although there are added expenses of moving snow and some difficulties with water supply in the winter months. Climatic conditions are not such as to cause major difficulties for year-round mining operations.

5.3 Local Resources

The company has a modern full-service camp on purchased land near to the property to provide a base for operations. Limited services are available in the village of Likely and supplies are generally brought to the property from Williams Lake.

5.4 Infrastructure

The main access route to the area is the Likely road, which passes north of the access road to the Imperial Metals Ltd. Mt. Polley copper-gold open pit mine, approximately 15 km to the southwest of the property. Power is available at Likely, with a major line in place to Mt. Polley. Water, subject to the usual constraints, is abundant in the area.

5.5 Physiography

The property covers an area north and south of Spanish Lake, from Mount Warren in the south to the high hill north of the lake. Topography is locally rugged with steep slopes and cliffs along deeply incised creek valleys. Elevations on the property vary from 930 m at Spanish Lake to 1,460 m near the top of Spanish Mountain and 1,325 m at the top of Mount Warren.

Vegetation is heavy forest, consisting primarily of thick stands of hemlock, balsam, cedar, and Douglas fir in the valley bottoms, with spruce, fir, and pine on the ridges. Locally, there are some cottonwoods and alders, and “devil’s club” is common. Underbrush is thick, especially in logged areas.

6 HISTORY

The following is a much-abbreviated synopsis of the history of the property and surrounding area up to 2008. For a much more complete treatment, refer to Singh (2008) and to Lustig and Darney (2006). The tabular summary below (Table 6-1) is adapted from Peatfield et al. (2009). Note that the historical records are not extensive, and are in some cases contradictory, especially in finer details. Table 6-1 is, however, a reasonable attempt to give an overall summary of early exploration activity on the property. Work conducted from 2009 to the present is described in Section 10 of this report.

Table 6-1: Summary of Historical (pre-2009) Exploration Activity at Spanish Mountain Deposit

Year	Company	Work Done
2008	Spanish Mountain Gold	40,449 m of diamond drilling in 161 holes Geological mapping, rock sampling, soil sampling
2007	Spanish Mountain Gold	26,993 m of diamond drilling in 126 holes Metallurgical testwork on drill core
2006	Spanish Mountain Gold	21,881 m of diamond drilling in 88 holes 5,009 m of RC drilling in 50 holes Geological mapping, rock sampling, soil sampling Airborne Geophysics and orthophotography on a property-wide scale
2005	Spanish Mountain Gold	7,746 m of diamond drilling in 35 holes 3,376 m of RC drilling in 30 holes Geological mapping, rock sampling, soil sampling
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 Gold 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 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 Mt. Polley.
1997	Wildrose Resources Ltd.	Wildrose gained 100% of the property through a plan of arrangement with Eastfield Resources Ltd.
1996	Cyprus Resources Ltd.	2,590 m of trenching signifying the first search for disseminated mineralization. 64 m of Trench TR 96-105 in the "Dodge Zone" assayed 0.716 g/t. This section is centred in the area of the bulk sample extracted in 1999
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
1993-1994	Renoble Mines	Set up a placer gold washing plant to recover gold contained in surficial material on the CPW claim (High Grade Zone)
1992	Eastfield Resources Ltd.	Consolidated the Spanish Mountain property
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) and 105 troy ounces were recovered.
1986-1988	Pundata Gold Corporation	37 diamond drill holes (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

Year	Company	Work Done
1986-1988	Pundata Gold Corporation	42 m of trenching in the LE Zone, 3 HQ diamond drill holes (267 m), 2 NQ diamond drill holes (157 m)
1987	Placer Dome Inc.	Optioned properties adjacent to the Spanish Mountain group (presently included in the Spanish Mountain property). 7 percussion holes (338.2 m) were drilled; significant gold values were encountered in overburden
1986	Mandusa Resources Ltd.	Optioned the north and southern areas of the property. Mandusa conducted geological mapping and IP surveys, and drilled 6 percussion drill holes (356.62 m)
1985	Mt. Calvery Resources Ltd.	7 diamond drill holes twinned RC holes. The diamond holes returned lower gold values than RC holes. Teck Corporation provided funding for this phase of exploration. 820 m of backhoe trenching (550 1 m channel samples), 29 RC holes (2,521 m) with encouraging results
1984	Mt. Calvery Resources Ltd.	Prospecting, geological mapping, rock, and soil sampling. 2,225 m of trenching, 10 diamond drill holes (467 m), 10 RC holes (589 m)
1984	Mt. Calvery Resources Ltd.	600 m of trenching and sampling, 7 RC holes (655 m)
1983	Whitecap Energy Inc.	Soil sampling (409 samples) with values up to 5,100 ppb. 100 m of trenching in 3 trenches
1983	Lacana Mining Corp.	Prospecting identified strong gold anomalies coincident with silicified argillite north of Spanish Lake
1981	Aquarius Resources Ltd.	Geochemical and geophysical program
1979, 1980 and 1982	E. Schultz and P. Kutney	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
1977-1988	LongBar Minerals	Two small programs
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 drill holes (792 m), 4 tons of handpicked ore shipped to the Tacoma Smelter
1938	NA Timmins Corp.	Overburden stripping, drove 2 small adits on large quartz veins
1933	Dickson and Bailey	Gold discovered in quartz veins on the NW flank of Spanish Mountain at 1100 m elevation
1921		Placer gold discovered in bench deposits on Cedar Creek

7 GEOLOGICAL SETTING

The following section was adapted from Peatfield et al. (2009).

7.1 Regional Geology

The Spanish Mountain deposit lies within the Quesnel Terrane, where this has been overthrust from the west onto the pericratonic Kootenay Terrane (Wheeler et al., 1991). At a broad overview scale, Wheeler and McFeely (1991) include the strata underlying the property in the Nicola Group alkaline arc volcanic and associated clastic sedimentary rocks of upper Triassic – lower Jurassic age.

In more detail, the area is adjacent to the Eureka Thrust (Struik, 1988) which marks the boundary between the Quesnel Terrane and the parautochthonous Barkerville Terrane of the Omineca Belt to the east (Struik, 1986). Historically, the rocks at Spanish Mountain have been correlated with the middle Triassic to early Jurassic sedimentary and volcanic rocks of the Quesnel River Group and Takla Group (Rees, 1981). Struik (1986) suggested correlation with the Nicola Group rocks to the south, which have similar age and lithology. The Crooked amphibolite is the basal unit of the Quesnel Terrane strata, and occurs discontinuously along the Eureka Thrust.

Bloodgood (1988) described the stratigraphy in the Spanish Lake area and correlated it with the rocks at Eureka Peak. Her stratigraphic sequence, from youngest to oldest, was:

- Volcanics – pillow lavas
- Volcanic wackes
- Tuffs, slates and phyllites (Unit 7)
- Grey phyllites and interbedded sandstones (Unit 6)
- Silty slates and phyllites (Unit 5)
- Triassic black phyllite – four mappable units correlated with rocks at Eureka Peak
- Crooked amphibolite – defines the base of the Quesnel Terrane.

The following summary of the geological setting of the Spanish Lake property is quoted from Panteleyev et al. (1996):

> *Studies in the map area, all within "Quesnel Terrane," confirm the presence of a regional synclinal structure formed within a Triassic continent-margin basin. It was infilled first with Triassic sediments and then Triassic to Jurassic volcanic rocks. Together these rocks constitute the Quesnel Trough. The basal lithologic units consist*

> *of mid-Triassic siliceous rocks to mainly younger pelitic, thinly bedded deposits with overlying, more massive volcaniclastic sediments. The younger epiclastic units pass upward or interfinger with Upper Triassic subaqueous volcanic deposits, mainly volcanic flows and breccia units. They are overlain, in turn, by subaqueous to subaerial Lower Jurassic volcanic flow and pyroclastic rocks and overlapping Lower to Middle Jurassic sedimentary assemblages. The volcanic rocks, and some Early Jurassic plutons, form the extensive magmatic edifice that defines the medial axis of the Quesnel island arc.*
>
> *The basal clastic rocks now form a continuous structurally complex black phyllite to metapelite unit along the eastern side of the map area. The rocks are well foliated at deeper structural levels but pass upward into weakly cleaved rocks. They are overlain by thick panels of the extensively block faulted successions. The basal sedimentary rocks are regionally metamorphosed to greenschist facies in the easternmost part of the map area. Metamorphic grade in the volcanic rocks is subgreenschist, consistent with burial metamorphism.*

Figure 7-1 is a fragment of a regional geology map, showing the general setting of the Spanish Mountain property. The black phyllite unit and associated strata (MuTrN on Figure 7-1) are host to the Spanish Mountain deposit.

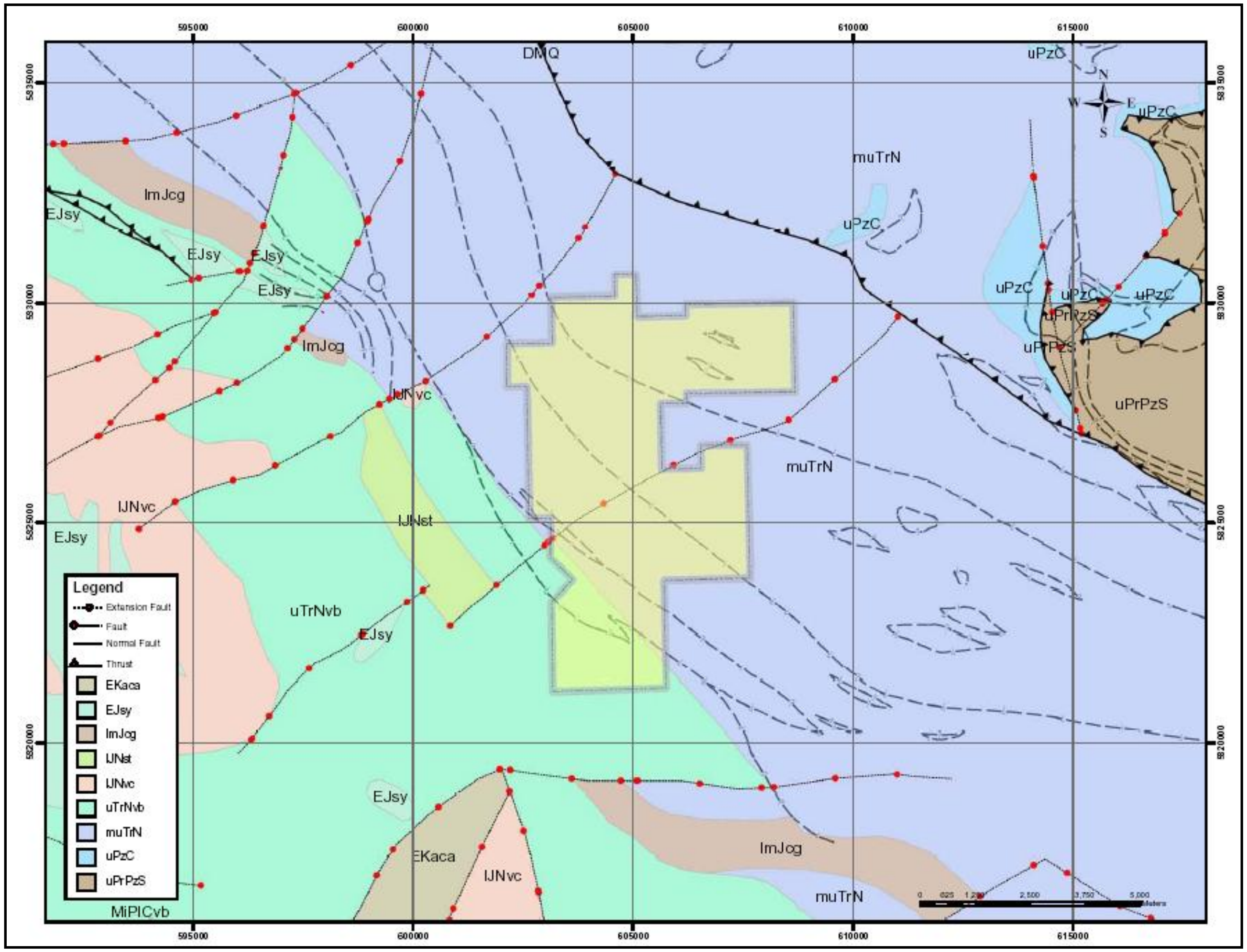
The region has been strongly affected by fold and thrust deformations; both features are readily identified in outcrop. Bloodgood (1988) summarized regional deformational features:

First phase structures (F_1) are recognized throughout the area and are represented by “mostly northeast-verging” folds of bedding (S_0). A penetrative slaty to phyllitic foliation (S_1) dipping shallowly to moderately to the southwest, is well developed axial planar to F_1 folds.

First phase structures are developed at all scales throughout the area. Small-scale isoclinal folds of bedding are pervasively developed, and there is evidence for larger scale overturned to recumbent folds. The structural vergence observed on mesoscopic F_1 folds, and the map pattern outlined by the exposures of Crooked amphibolites south of the Cariboo River, indicate a large antiformal culmination in this area, which is interpreted as a large, easterly verging Phase 1 nappe structure.

Phase 2 structures (F_2) overprint and refold F_1 structures throughout the area. Structural elements associated with F_2 are well-developed non-penetrative cleavage (S_2), manifest as a crenulation cleavage, or a spaced cleavage, or locally a fracture cleavage.

Figure 7-1: Regional Geology Map



F_2 folds occur as open, buckle folds and conjugate kink-type folds. The axial surfaces of F_2 structures are present as conjugate sets dipping moderately to the northeast and southwest. Phase 3 deformation is ubiquitous throughout the area as a spaced cleavage and fracture set. Numerous steeply dipping, northeast-trending normal faults have been recognized within the volcanic sequences to the west of the study area; these faults post date regional folding. High angle faults are recognized.

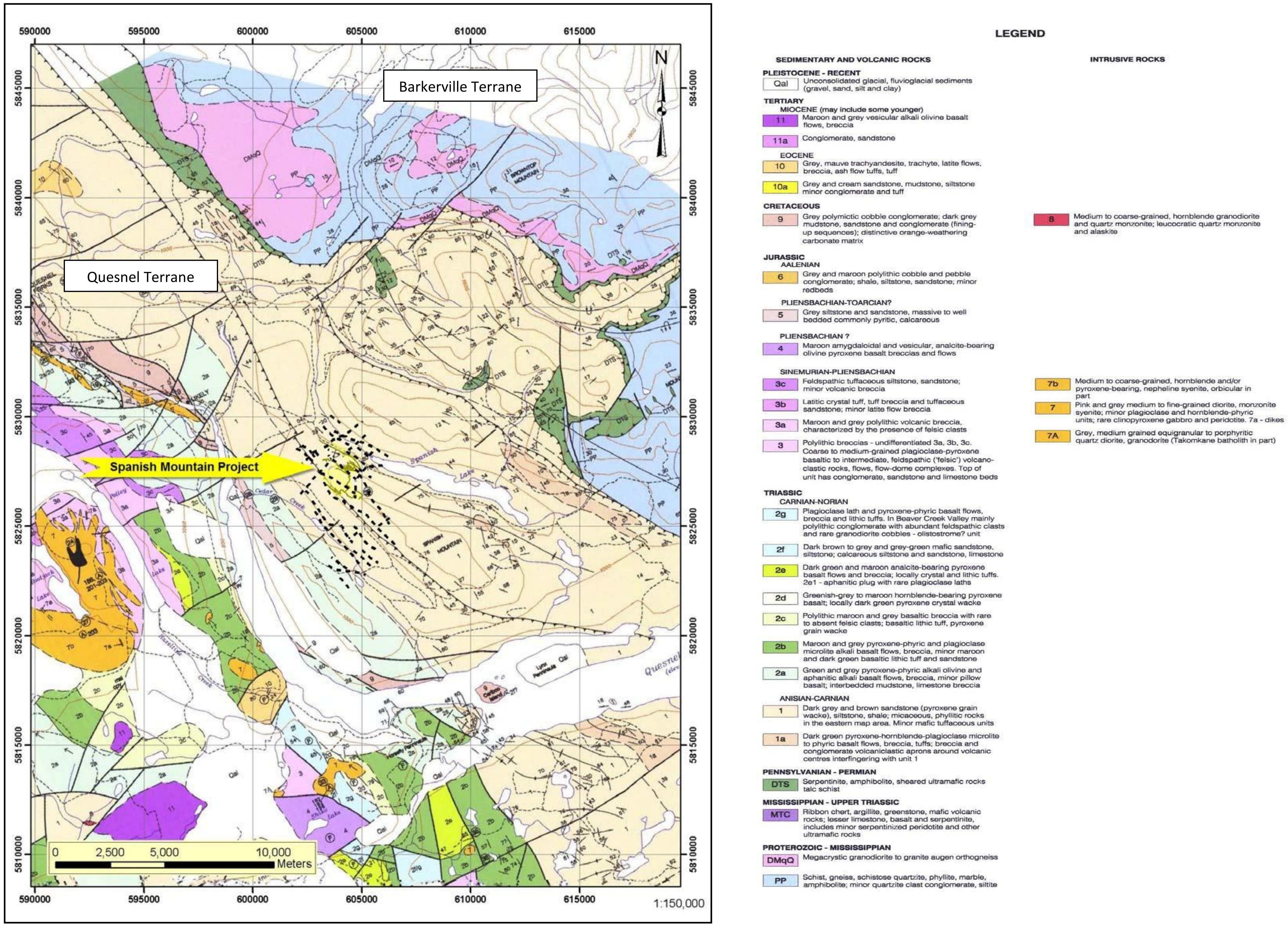
Several thrust faults are mapped in the area, the most prominent being the Eureka thrust, a southwesterly dipping fault, which separates the Quesnel Terrane from the Barkerville Terrane. The Eureka Fault is mainly a topographic feature in the Spanish Mountain area and no surface exposures of the Eureka thrust have been noted, either in the literature or on map data. Detailed mapping by Rees (1979) (Figure 7-2) as part of a Ph.D. thesis project outlined several kilometres of thrust faulting in the region. Cross-section data from this thesis indicated that the Eureka Fault is a folded thrust fault (affected by D_2 deformation); however, the map data do not agree with this interpretation. Singh visited several localities on this map sheet; owing to excessive vegetation and low-lying areas, he could not verify the map data.

Figure 7-2 shows a more detailed map of the general region, based on the work of Rees.

Regional scale alteration consists of minor iron-carbonate alteration with local pervasive coarse-grained ankerite, which is more prominent near fault zones. Black shale sequences tend to have trace amounts of pyrite, which may be diagenetic, and which is locally recrystallized due to metamorphism or hydrothermal activity.

Several styles of mineralization have been documented within the Quesnel Terrane in this region. The most prominent is the alkalic copper-gold porphyry-style mineralization such as that at the producing Mt. Polley Mine. The QR deposit (Minfile number 093A121 – classified therein as Au skarn) is located some 25 km west-northwest of Spanish Mountain. Other styles include Cu-Zn massive sulphides in limey quartzite (Sellers Creek – Minfile number 093A 131 – classified therein as “Besshi type”); various gold vein occurrences; and placer gold deposits. The Spanish Mountain deposit is one of a number of similar sediment-hosted vein and disseminated gold deposits in the region.

Figure 7-2: Regional Geology Map Modified after Rees (1979)



For more details of geology in the general region, readers are referred to papers by: Rees, (1981), Monger et al. (1982), Struik (1986 and 1988), and Bloodgood (1987 and 1988). At present, there are many unanswered questions about the regional mapping in the immediate Spanish Mountain area. Work conducted by Rees, Bloodgood, Struik, and others is without question competent, thorough, and of high quality; however, with the addition of detailed mapping and over 60,000 m of diamond drilling at Spanish Mountain, much of the regional geological data are brought into question. In particular, structural and stratigraphic relationships documented at Spanish Mountain do not correlate well with the regional geological or structural understanding.

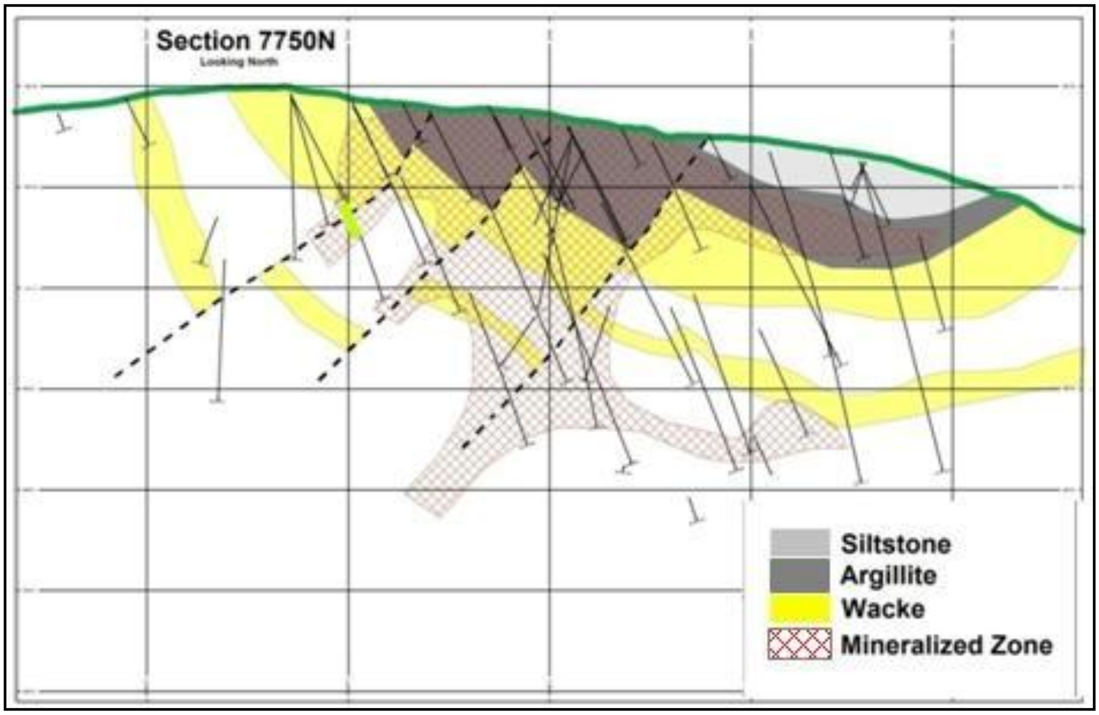
7.2 Property Geology

The Spanish Mountain property is typically overburdened by up to 10 m of glacial deposits consisting of gravels, sand, till, and local colluvium. Outcrop exposures on the property consist mainly of sedimentary rocks (phyllite, argillite, shale, wacke, conglomerate, and siltstone), minor volcanic facies (mafic volcanics, pillow basalts), and minor intrusive (quartz-feldspar porphyry). Heavy overburden cover impedes geological map interpretation for much of the property. Sub-surface information from diamond and RC drilling along with airborne geophysics have aided greatly in the geological interpretation of the property. Strong evidence of faulting and folding exist in both mapping and drilling. Folds are typically isoclinal and open "warps." Faults are manifest as thrusts and as normal and strike-slip faults.

7.2.1 Geological Model

During the fall of 2008, reinterpretation of the structure and geological model of the Main Zone at Spanish Mountain led to the recognition of three sub-parallel faults striking approximately 10 to 35 degrees and dipping shallowly (35 to 45 degrees) to the west. These faults are among a series of structures identified in drill core and at surface, which occupy a corridor approximately 400 m wide x 2 km long. Within this corridor, it appears that these faults normally offset stratigraphy by as much as 100 m (downdropped on the west side). The amount of offset is dictated primarily by rock type whereby the more competent units such as wacke and siltstone exhibit clear offsets and the more plastic units such as argillite manifest the offset by folding. Previous interpretations had recognized the apparent offset of units along these faults but had interpreted them as F4 folds. The current interpretation includes both folding and faulting along these structures, making the structures axial planar to F4 folds. Figure 7-3 is a schematic east-west cross-section showing the interpreted faults and related offsets in stratigraphy.

Figure 7-3: Fault Zones within the Main Zone with Offset Stratigraphy – Section Looking North



In addition to affecting stratigraphy, these faults may have also acted as permeable conduits for mineralizing fluids, as gold mineralization appears to be concentrated around these structures. At present, the main fault (termed the M2C Fault) extends through the centre of the deposit for approximately 2 km. All of the gold mineralization in argillite horizons (Upper, Lower, and North Zones) occurs within a 400 m wide corridor surrounding this central fault. Units which are less permeable or do not provide a favourable chemical "trap," such as siltstone and greywacke, tend to have narrower zones of gold mineralization near the faults, whereas the argillite sequences have wider spread disseminated gold.

At surface, the M2C Fault is best recognized in the Imperial Metals Pit where visible gold is often associated with quartz veins within the fault. Several other faults have been observed at surface, trending north to northeast. None of these appears to have any strike-slip offset, nor do they separate structural domains.

During the 2007-2008 field season, several faults were noted in areas where drilling was very difficult, particularly in the eastern and northern parts of the Main Zone. It would appear that these areas are associated with sections of stratigraphy that are steeply dipping to the northeast. It is likely that the drill holes were lost because they were drilling down-dip along argillite beds instead of drilling into structures. The lost holes are also a result of drill contractor inexperience; several previously "lost" holes were re-entered or re-drilled with >95% core recovery to the target depth with a more experienced contractor.

The current geological model simplifies previous fault-and-fold models and potentially accounts for the localization of gold mineralization within the Main Zone. Stratigraphy in the Main Zone is well documented and occurs within a single overturned (based on graded bedding) limb of an F_2 fold. This fold has been modified by F_3 and F_4 structures, causing wrinkling of the surface and large-scale open warps. Units generally strike northwest southeast and dip between 60 and 25 degrees to the northeast. The average dip of units is approximately 45 degrees. There is an apparent flattening of units in the centre of the Main Zone, which may represent a large-scale F3 fold hinge zone.

7.2.2 Stratigraphy

Drilling and geological mapping have identified several key stratigraphic relationships in the Main Zone. Stratigraphy outside of this Zone is poorly understood. Geological mapping is only useful to discern wacke and siltstone units, as individual argillite beds tend to be folded and nearly impossible to correlate. Figure 7-4 shows the large wacke and siltstone areas mapped at surface. Re-interpretation of drill core data late in 2008 has put into question some of the contact relationships observed on this map; further work was recommended to refine geological mapping.

Alteration has obliterated many primary textures particularly in wacke sequences. It is therefore necessary to use geochemistry and petrography to aid with descriptions and correlation within this unit. Fortunately, the altered sections of wacke have a very distinctive geochemical signature (Mg, Mn, K, and Al) and can be correlated with confidence on cross-sections. Table 7-1 details the salient characteristics of various units.

7.2.3 Other Lithologic Units

Outside of the Main Zone sequence, stratigraphy is poorly understood. Poor surface exposures and limited diamond drilling make correlation virtually impossible at this stage in exploration. The following lithological units are recognized but their stratigraphic relationships are poorly understood.

- Amygdaloidal basalt:
  Fine to medium-grained, strongly altered (buff brown to pale colour), strongly amygdaloidal with abundant scoria textures throughout. The unit is intersected in only one hole and its down hole depth extent is >100 m. This unit is strongly altered (carbonate and sericite) and mineralized, with several quartz veins with galena, sphalerite, and chalcopyrite. Contacts with overlying argillite and siltstone are faulted. This rock was intersected in diamond drilling in the “placer area.”

Figure 7-4: Property Geology Map – White Areas Unmapped

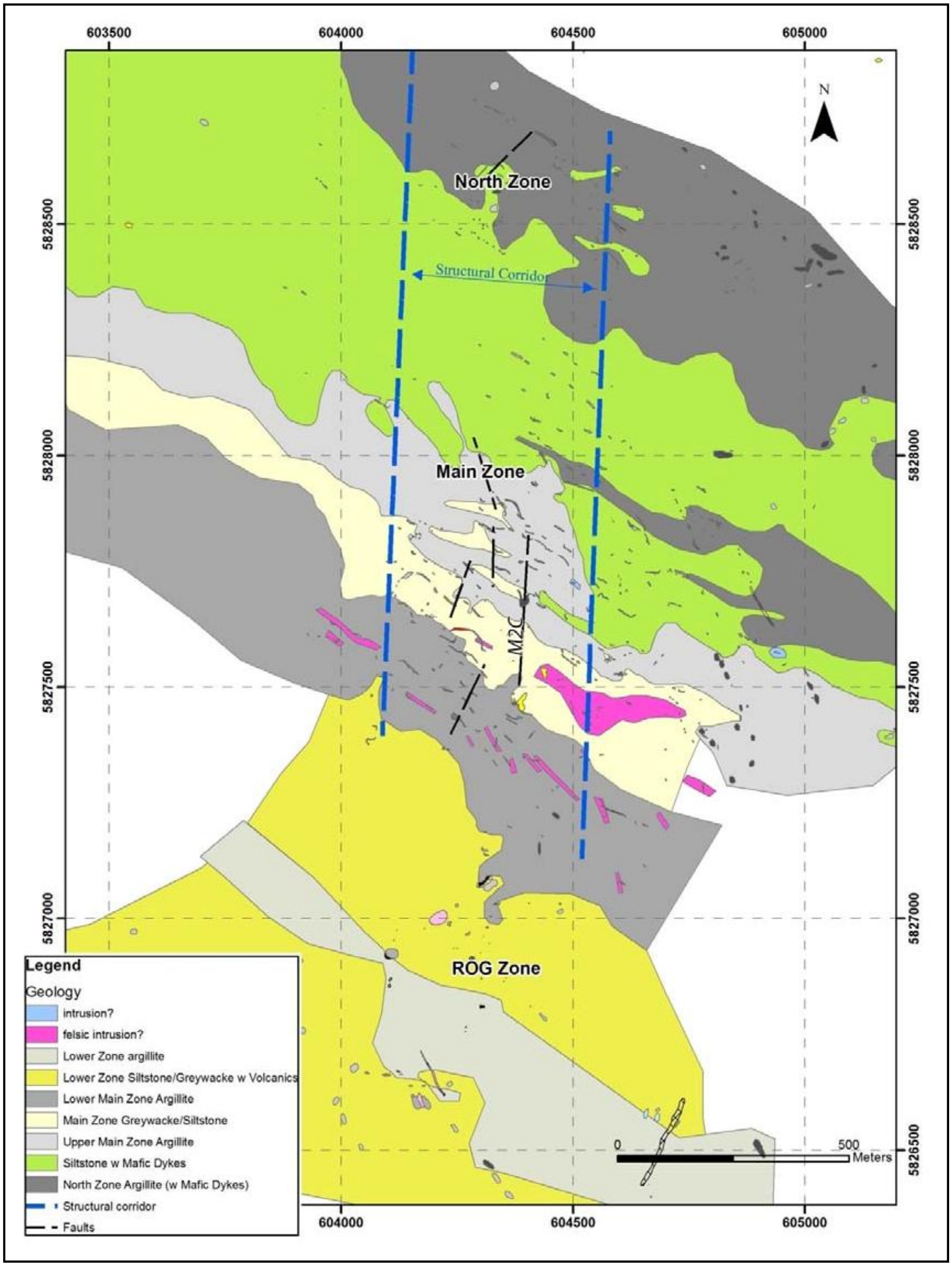
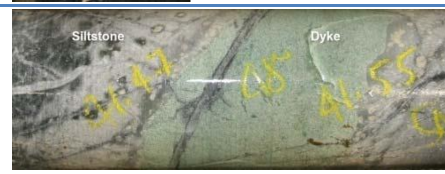


Table 7-1: Characteristics of Various Rock Types

Unit	Description	Example
North Zone Argillite	Fine-grained black argillite with siltstone interbeds, generally 30–100 m thick. Large interbeds or boudins of wacke up to 30 m are common. This unit can host wide zones of disseminated gold mineralization. This unit is typically iron-carbonate altered, sericitic, and quartz veined. The structural base of this unit is poorly understood as few drill holes have penetrated it. The unit generally has 1–3% pyrite as fine disseminations and coarse-grained cubes.	
Upper Siltstone (with mafic dykes)	Medium to light grey, finely laminated, rarely interbedded with argillite and wacke. Unit is up to 130 m thick. Several hornblende phyric mafic dykes are noted in this unit, decreasing in volume with depth. Visible gold has been noted in quartz veins in several locations in this unit. Alteration consists mainly of sericite (chromium rich), iron carbonate or high Mg dolomite, silicification, and strong local quartz veining. The lower contact is often faulted and on rare occasions is conformable.	
Main Zone Argillite	Black to dark grey, locally finely laminated with soft sedimentary features. Some sections are phyllite with graphite on fracture surfaces and the unit is up to 100 m thick. Interbeds of argillaceous conglomerate (near the structural top) are common. The upper section of this unit tends to be "fragmented," a term adopted at Spanish Mountain to refer to the clastic nature of the unit most likely occurring early in the tectonic history. This unit will have rare wacke interbeds and mafic dykes. The bulk of the disseminated gold mineralization of the Main Zone (>65%) is hosted in this unit. The lower contact is either faulted or conformable.	
Lower Wacke	Fine to coarse-grained sequence of wacke (quartz wacke)/siltstone and local argillite. Rare sections of feldspathic wacke are observed. In many instances, this unit is a graded sequence (fine direction down hole) over 10s of metres. Pervasive alteration makes identification of this unit in hand specimens nearly impossible. Geochemistry and petrography have assisted greatly in interpreting and correlating this unit. Several occurrences of visible gold have been noted in quartz veins in this unit.	
Lower "altered" wacke	Fine to coarse-grained light grey colour, often appearing bleached. Some sections are quartz-rich (2–3 mm detrital grains) and have occasional feldspar grains. Fine laminations can be observed, although most primary textures are destroyed by sericitization and iron-magnesium carbonate alteration. Occasional albitized zones are recognized. Some sections are graded and conformable with the lower contact. Occasional fault contacts are observed.	

Unit	Description	Example
Conglomerate	Fine to coarse, angular to sub-rounded graded (fining down hole) sequence. Clasts are dominantly siltstone, wacke, and occasional argillite. This is a fining down sequence generally conformable with the lower contact.	
Lower Argillite	Black to dark grey, interbedded argillite/siltstone and wacke. This unit may contain up to 30% wacke interbeds or boudins. The unit may be locally fragmented. Core angles in this unit are between 50 and 80 degrees and generally consistent, in contrast to the upper argillite which will have strongly contorted bedding and highly variable core angles. This unit exhibits strong iron-magnesium carbonate alteration and has local sericite alteration. Pyrite is generally less than2%.	

- Quartz porphyritic rhyolite:
  Intersected in fewer than five drill holes, this unit is fine grained with 2–3% of 1–1.5 mm quartz phenocrysts and tends to be <20 m thick. Stratigraphic relationships of this unit are poorly understood.
- Diorite:
  Medium-grained, weakly chloritic diorite (intersected in one RC drill hole north of the Main Zone).
- Quartz Feldspar Porphyry:
  First mapped by Wildrose geologists approximately 4 km southeast of the Main Zone, this rock has since been intersected in the southern ROG holes at the top of Spanish Mountain. It is a coarse grained (>3 mm) quartz and feldspar phyric porphyry.
- Volcanic Tuff:
  Variable fine to coarse-grained tuff with very angular lithic and crystal fragments (generally mafic in composition); this unit may be more common in the sequence as it may often be interpreted as “volcanic wacke.” The distribution of this unit is poorly understood and it typically occurs within large sequences of wacke or in the lower argillite of the Main Zone.
- ROG Porphyry:
  Intersected in ROG drill holes. Light to medium grey with 5–10%, 1–5 mm subhedral to anhedral zoned feldspar phenocrysts in a felsic groundmass consisting of up to 50% translucent quartz, 45% white feldspar, and 5% mafics (probably biotite). Moderately to strongly altered.

7.2.4 Structure

Given that the property lies within the centre of a major fold and thrust belt, it is expected that regional structures would be mimicked at the local scale. Mapping to date has recognized up to four phases of fold deformation with associated faulting. Recent structural interpretations have recognized north-south to northeast-trending, west-dipping fault zones that appear to normally offset stratigraphy. This offset has been associated with the last phase of deformation; however, some interpretations indicate that it may have been affected by earlier (D_2/D_3) deformation, and is therefore a remnant of an earlier deformation event. These deformation stages are summarized as follows.

- D_0/D_1 Deformation
  Manifest as a bedding parallel foliation fabric, often folded into D_2 folds. Regionally this stage of folding should be evident as northeast-verging tight isoclinal folds; however, such vergence has not been noted on the property. It is suggested that the foliation fabric may be a simple compaction fabric (D_0).
- D_2 Deformation
  Tight isoclinal folding manifest as folded bedding and foliation planes. Bedding cleavage relationships on the property suggest a northeast-verging fold system. Figure 7-5 shows bedding measurements and fold axis plunging shallowly to the southeast. Figure 7-6 shows D_2 folds in outcrop. D_2 foliation (S_2) is generally sub-parallel to bedding and S_1 fabric, and is therefore difficult to measure except in fold hinge domains. F_2 folds strike between 130 and 140 degrees, and dip between 30 and 50 degrees to the southeast and northwest.
- D_3 Deformation
  D_3 deformation manifests itself as open warping of D_1/D_2 fabrics and folds. S_3 foliation is a widely-spaced cleavage which is nearly vertical. D_3 deformation may be coincident with D_2 deformation, as they are both influenced by the same tectonic forces and have sub-parallel axial planes. Figure 7-7 shows bedding planes modified by D_2 and D_3 deformation events; in the figure, F_2 fold axes have been re-folded by F_3 (green axial plane). Bedding (red colour) is strongly folded and disrupted. The D_3 event locally warps stratigraphy on a 10 to 20 m scale; it may also be responsible for the overall flattening of the Main Zone stratigraphy on a 500 m scale.
- D_4 Deformation
  D_4 deformation is the least recognized on the property and is represented as large scale (3 m to 4 m) warping of bedding and foliation fabrics. This warping occurs perpendicular to D_1-D_3 with no discernible axial planar cleavage or fold plunges. D_4 deformation may explain doubly-plunging folds (Figure 7-7). The trace of the axial plane for four folds would strike approximately 10 to 35 degrees and dip shallowly to the west at 35 to 45 degrees. This axial plane forms central fault zones that are believed to be associated with gold mineralization and normally offset stratigraphy (Figure 7-3).

Figure 7-5: Stereonet Plot showing Bedding and Fold Axes

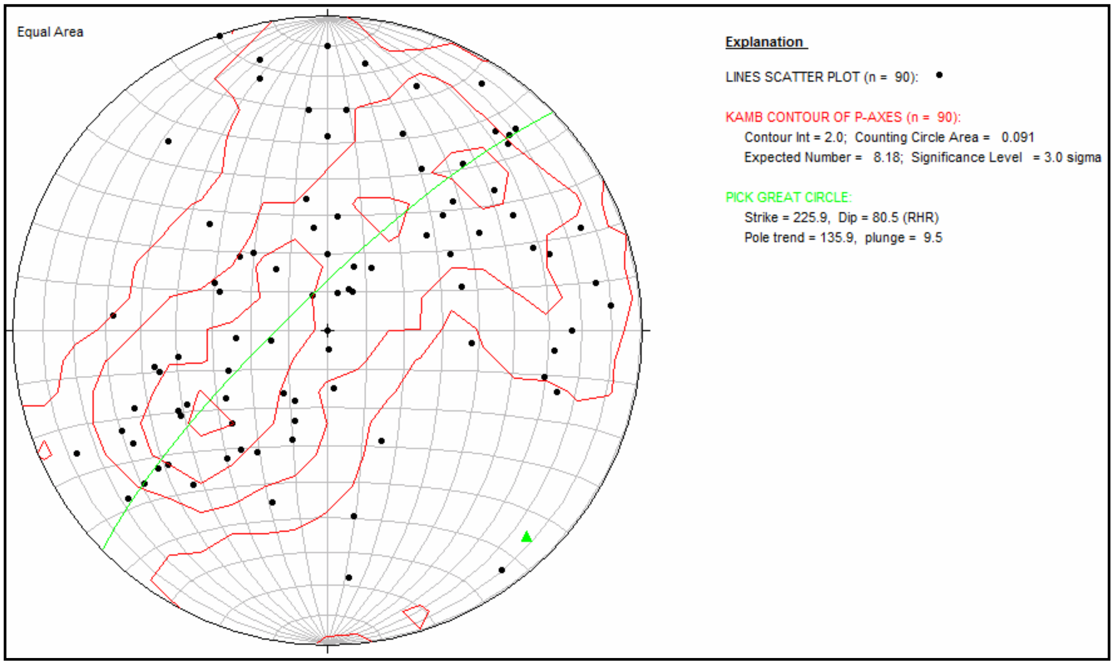


Figure 7-6: D2 Folds in Outcrop, Modified by D3

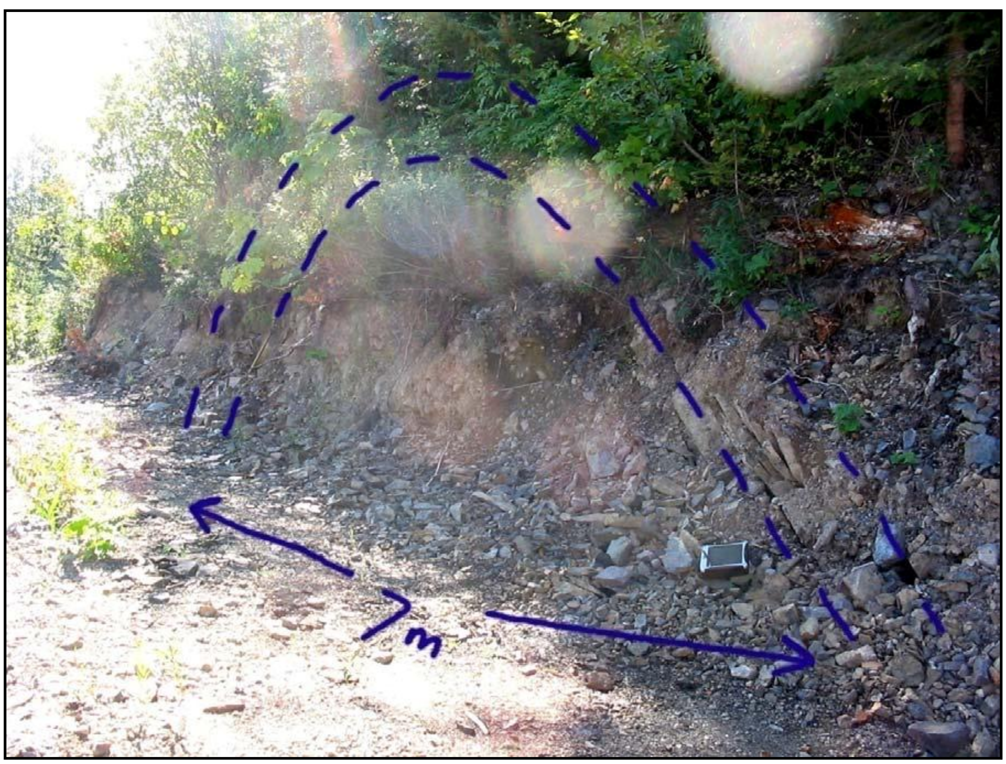
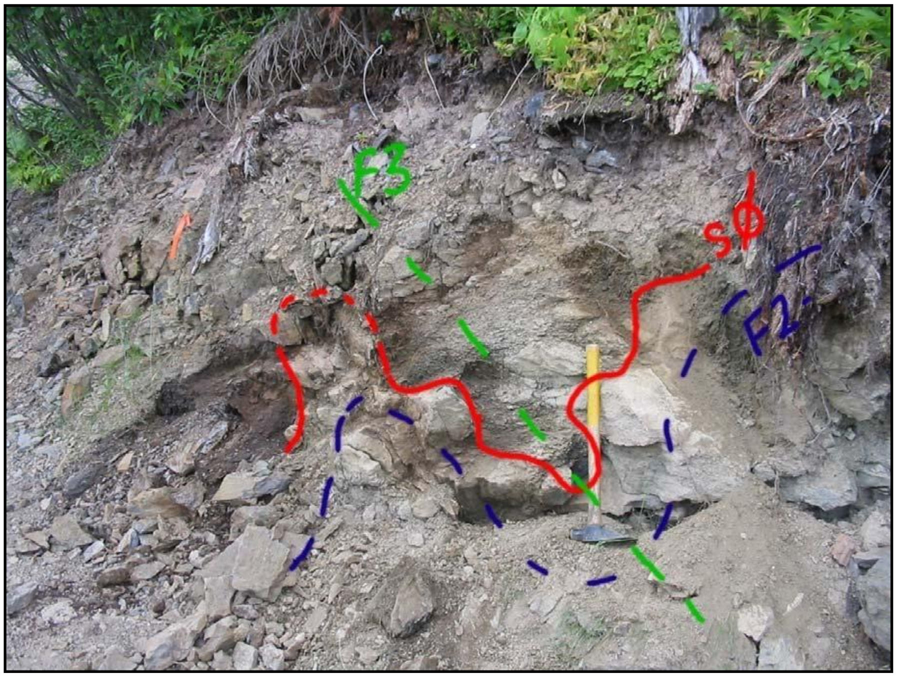


Figure 7-7: D_2 and D_3 Deformation



7.2.5 Alteration

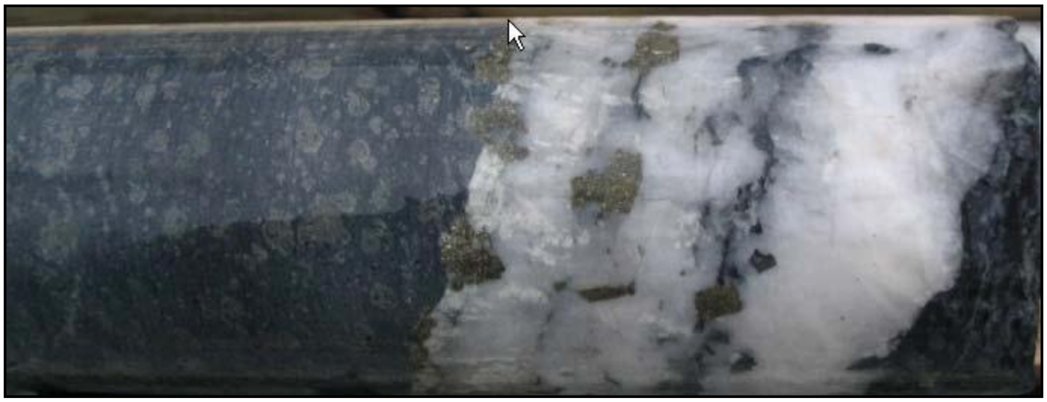
Alteration on the property is generally very widespread and consists mainly of iron-magnesium carbonate and sericite assemblages. Minor sections of silicification and albitization are observed. Sericite content ranges in the various units between 5% and 45% (Ross, 2006a) locally up to 60%. Fine-grained rutile is also present in most lithologies as fine disseminations and may be related to S_3 crenulation cleavage (Ross 2006a). Fuchsite/mariposite (Cr-bearing mica) is present mainly in wacke sequences and mafic dykes. Ross (2006b) indentified chrome-bearing spinel in petrographic work as a possible source for the chrome.

Carbonate alteration occurs mainly as iron-magnesium carbonate (Ross, 2006a). Several phases of carbonate alteration are present. Ross stated:

The earliest stage of carbonate occurs as ragged, rounded porphyroblasts, overgrowing the S_1 fabric. These porphyroblasts are often partially to intensely oxidized. The second stage porphyroblasts occur as generally well formed rhombs. These rhombic crystals overgrow the older carbonate. S_3 kink bands wrap around these porphyroblasts. Where this alteration is most intense, the carbonate is less rhombic, and becomes pervasive interlocking patches, sometimes preserving the underlying rock texture. Carbonaceous matter is often incorporated into the *porphyroblasts, but is remobilized into dendritic patterns, destroying the foliation texture.*

In hand specimens, carbonate porphyroblasts can be up to 10 mm and often give the rock a "knotted" appearance. It is worth noting that "knotted phyllite" is the host rock for the nearby Fraser Gold deposit currently being explored by Hawthorne Gold Corporation. Figure 7-8 shows an example of carbonate porphyroblasts.

Figure 7-8: Large Carbonate Porphyroblasts in Siltstone (to the left of the quartz vein)



Sericite alteration (Figure 7-9) occurs pervasively throughout most units and can be very coarse-grained; some sections may contain paragonite (lacking potassium). In the finer-grained siltstone and argillite, sericite tends to align to foliation; however, in more competent wacke sequences it does not appear to be affected by foliation. A plot of molar ratios of K/Ti vs. Al/Ti (Figure 7-10) shows a good correlation between gold mineralization and sericite alteration.

Pyrite is ubiquitous throughout argillite units, with typical values ranging between 1% to 1.5% and locally up to 5%. Rare sections of 20% to 35% pyrite have been observed. Pyrite occurs as fine disseminations, large 0.5 cm to 2 cm cubes, and along vein and fracture margins as blebs. Workers at the University of Tasmania (personal communication, Professor R. Large) have identified up to three stages of pyrite growth, the last two being metamorphic. In hand specimen it is virtually impossible to discern these stages. Pyrite crystals tend to be larger in wacke units and locally crosscut foliation planes suggesting a very late stage re-crystallization.

Figure 7-9: Strongly Sericitized Greywacke, with Visible Gold in a Quartz Vein

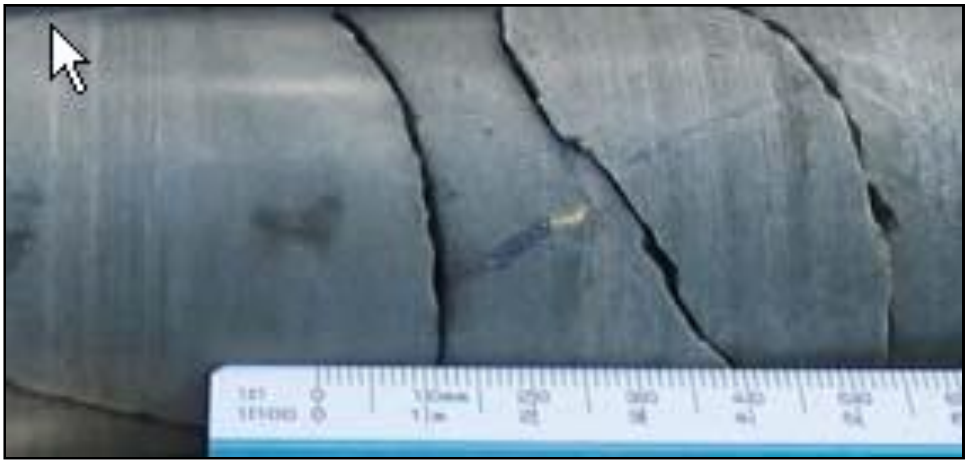
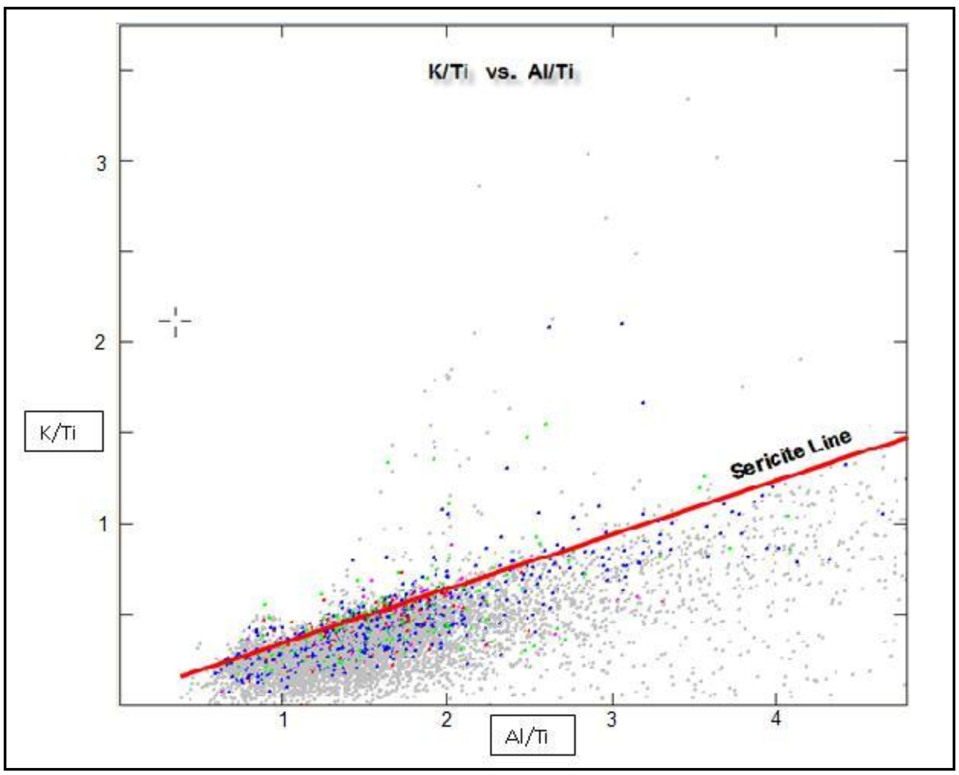


Figure 7-10: K/Ti vs. Al/Ti Ratio Plot[3]



[3] Note that gold values tend to cluster around the sericite line. Coloured values are >0.20 g/t; red values are >1.0 g/t Au.

8 DEPOSIT TYPES

The following section was taken from Peatfield et al. (2009)

Historically the area has been explored for lode gold hosted in quartz veins. Vein paragenesis suggested the veins are late in the tectonic history and can contain very high grades of gold. The region has also been extensively explored for placer gold deposits, “besshi-type” massive sulphide deposits and alkali copper-gold porphyry deposits. At present, the following deposit types are recognized in the region:

- Placer gold (surficial deposits including colluvium)
- Alkalic porphyry copper-gold (Mt. Polley type)
- Besshi-type massive sulphide
- Disseminated or orogenic gold
- Alkalic skarn gold (QR Mine).

The main exploration target in the past at Spanish Mountain has been auriferous quartz veins. To date, the highest assay values have all been attributed to quartz veins.

At present, the Spanish Mountain property is classified as an orogenic gold deposit, or a Sediment Hosted Vein (SHV) deposit as defined by Klipfel (2005). SHV deposits share many common characteristics such as host rocks, tectonic setting, mineralization style, trace element geochemistry, and hydrothermal alteration. Klipfel (2005) wrote:

> *[The term SHV] is applied to a family of deposits that occurs throughout the world but are poorly known and understood. They are most prolific in both size and number in Asia. Many are in the former Soviet Union with geologic, geochemical, and geophysical information usually in Russian and difficult to obtain. The work here is based on personal visits to some of these deposits and review of available reports in English and Russian. Included in this group of giant gold deposits are Muruntau (80 Moz), Sukhoy Log (20 Moz), Amantaytau, Daugiztau, Kumtor, Bakirchik, Olympiada, Nezhdaninskoe, Natalka,and Maysky in Asia. In Australia, the numerous deposits of the Victorian gold fields include Bendigo, Ballarat, Fosterville, and Stawell. In New Zealand, the Otago Schist Belt hosts Macraes Flat and numerous small deposits. In South America, pre-Cordillera rocks of Peru, Bolivia, and Argentina host many small to medium deposits with past production from pre-conquest time. In North America, numerous small to medium deposits occur in the Meguma Terrane of Nova Scotia and in the southern half of the Seward Peninsula, Alaska.*

These deposits are united as a group by having in common their tectonic setting, host rocks, alteration style, metal content, fluid chemistry, and to some extent by the absolute and relative timing of formation.

The Spanish Mountain property shares many characteristics with other SHV deposits in that it hosts wide zones of >1.0 g/t Au in argillite/siltstone and wacke sequences. The contact relationship between the wacke and argillite/siltstone is an important control on localizing gold mineralization. This contact relationship was observed by Singh at Natalka in the Magadan Region, Russia, in 2007, where a significant (>60 Moz) gold deposit has been defined. The deposit occurs along a contact between black shale sequences and a lapilli tuff unit. It is worth noting that the highest grades at Natalka are found in the competent lapilli tuff unit. In comparison, the highest individual core samples at Spanish Mountain also occur in the competent wacke sequence.

An alternative definition for the Spanish Mountain deposit could be an "orogenic gold deposit," using the terminology of Groves et al. (1998) and Goldfarb et al. (2005). In summary, such deposits are characterised by gold contained in quartz, carbonate vein networks or disseminated within metamorphosed sedimentary, and volcaniclastic rocks, in many cases with no obvious plutonic rock association.

Spanish Mountain shows many of the features common to these deposits; perhaps the most important difference is the low observed levels at Spanish Mountain of base metals, and the paucity of trace elements such as arsenic and antimony, which are common to many of the deposits of this type.

9 MINERALIZATION

The following section was taken from Peatfield et al. (2009)

Past work has identified free gold in quartz veins and disseminated in black shale (argillite), and free gold occurring in quartz remobilized into fold hinges (personal communication with R. Mickle). The property has historically produced less than 1,000 oz of gold from hard rock in small pits, in addition to modest amounts of placer and colluvial gold. Various prospectors and miners throughout the years have collected large nuggets and “pieces” of gold measuring 10 cm across. Several >3 mm pieces of gold have been observed in drill core. Gold mineralization has been noted in the following styles:

1. free gold in quartz veins
2. free gold in fractures in pyrite
3. free gold in base metal sulphides (sphalerite, chalcopyrite, galena)
4. gold associated with arsenopyrite
5. disseminated gold in black argillites
6. disseminated gold in fault structures often associated with quartz veins.

By far the most significant styles of mineralization are numbers 5 and 6. These styles of mineralization have been traced for over 2 km in strike length and occur in multiple stratigraphic horizons. Recent interpretation places this disseminated mineralization in close proximity to a major north-south trending zone of faulting (Figure 7-4).

There are a few occurrences noted of elevated gold contents associated with fault zones. In many cases, the fault zones contain quartz veins, which may be the hosts for the gold. It is not clear at this time what influence the fault structures might have on the overall gold content of the deposit.

There is an overall paucity of Cu, Zn, Pb, As, Sb, and other trace metals in the system and as such, gold is the only pathfinder element.

Figure 9-1 is a schematic long section over the deposit, and Figure 9-2 is a stratigraphic column demonstrating North, Main, and Lower Zone stratigraphy. Both of these diagrams demonstrate the relationship between mineralization and lithological units.

Figure 9-1: Schematic Long Section showing Mineralization and Lithologies

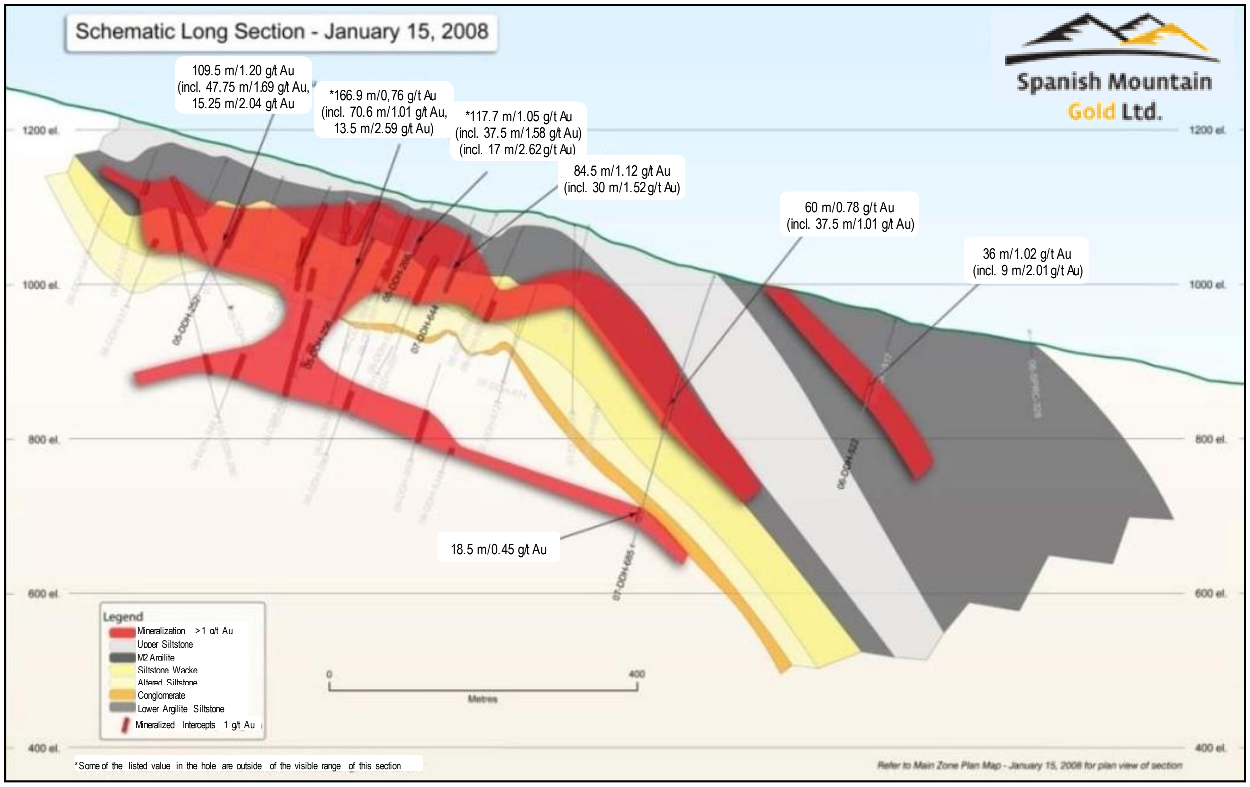
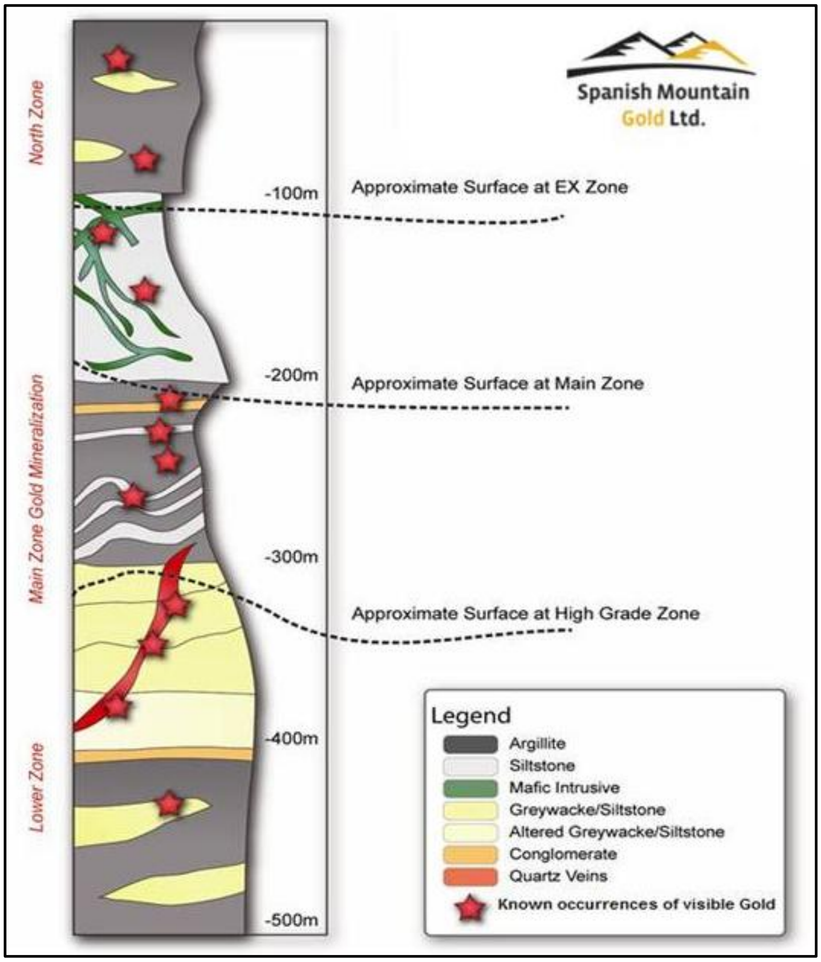


Figure 9-2: Composite Stratigraphic Column – North, Main, and Lower Zones



9.1 Gold in Quartz Veins

Although a minor component, quartz veins with free gold have produced the highest-grade individual samples on the property. For example, hole 07-DDH-588 intersected 241 g/t Au over 1.5 m. These veins tend to occur only in more competent facies such as wacke and siltstone; the veins are discontinuous at surface and exhibit a strong gold nugget effect. Analytical work suggests that along with coarse gold, there is a significant portion of fine gold, which also exhibits a nugget effect.

The economic significance of this style of mineralization may only be realized in a mining scenario. Currently these veins have been followed with confidence for approximately 40 m. Within these 40 m, it is often difficult to reproduce bonanza grade results. Gold is often associated with base metals in these veins; it is common to identify sphalerite, galena, and chalcopyrite with gold. Geochemically, the base metals remain insignificant; however, they are a good indicator of gold mineralization. Work conducted at the University of Tazmania in 2006 and 2007 (personal communication, R. Large) would suggest that gold and base metals may have been remobilized into these veins.

These veins typically appear to have been emplaced late in the tectonic history and crosscut all foliation fabrics. Through oriented core data and geological mapping, it is determined that the veins generally strike between 010 and 050 degrees and dip at various angles to the southeast and northwest. One set of flat lying veins dipping at approximately 10 to 35 degrees at a strike of between 70 to 100 degrees has been identified. Several "blowouts" — vein sections, which are 1 m to 5 m in thickness – have been identified on the property.

Several attempts have been made to correlate veins with overall gold grades in areas where gold mineralization is widespread. The data to date are inconclusive, as there does not appear to be a strong correlation between quartz veins and gold mineralization, particularly in argillite sequences. Several domains of intense veining (3–5 veins per metre) have been documented in drilling; however, these do not contain significant gold values. For example, section 1625N in the Main Zone demonstrates how quartz veins and gold values relate. In this example, Hole 05-DDH-255 intersected 45.5 m of 1.63 g/t Au within the Main Zone argillite. This interval contains five quartz veins. In contrast, Hole 07-DDH-595 intersected 20 quartz veins in argillite and does not contain any significant gold values.

To date, over 40,000 individual quartz veins have been recorded in the database. Of these, 2,403 veins occur within samples with over 1 g/t Au or 14.6%. Conversely, 16,460 veins or 40.5% are in samples with gold values below detection (<0.003). There appears to be a linear relationship between gold values and angles to core axes (Figure 9-3), and an inverse relationship between vein size and gold values (Figure 9-4).

9.2 Disseminated Gold Mineralization

By far the most significant gold mineralization at Spanish Mountain is hosted in wide zones (10 m to 135 m) within argillite/siltstone and lesser wacke sequences. This mineralization typically occurs near contacts with competent greywacke host rocks and often spans the contact and occurs proximal to north-south trending fault zones. This style of mineralization, although most widespread, is the least understood on the property. Several attempts at gold characterization have been undertaken by Ross (2006b), by workers at the University of Tasmania, and by Singh (2008). All attempts have failed to explain how the majority of disseminated gold occurs in the rocks.

The following quote from Ross 2006b) was based on examination of 15 samples of the disseminated mineralization type:

> *In conclusion, native gold (electrum) [sic; probably argentian gold] was identified in four samples, and it occurred as inclusions and fracture-fill in pyrite, on crystal boundaries between pyrite crystals and in the gangue adjacent to pyrite. It is very fine grained <20 µm, and generally <5 µm. It is associated with equally fine-grained chalcopyrite-galena-sphalerite, which occurs in all the same habits. All of the mineralized samples occurred in variably carbonaceous mudstone/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 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.*

Figure 9-3: Sample Grade of Gold vs. Quartz Vein Angle to Core Axis

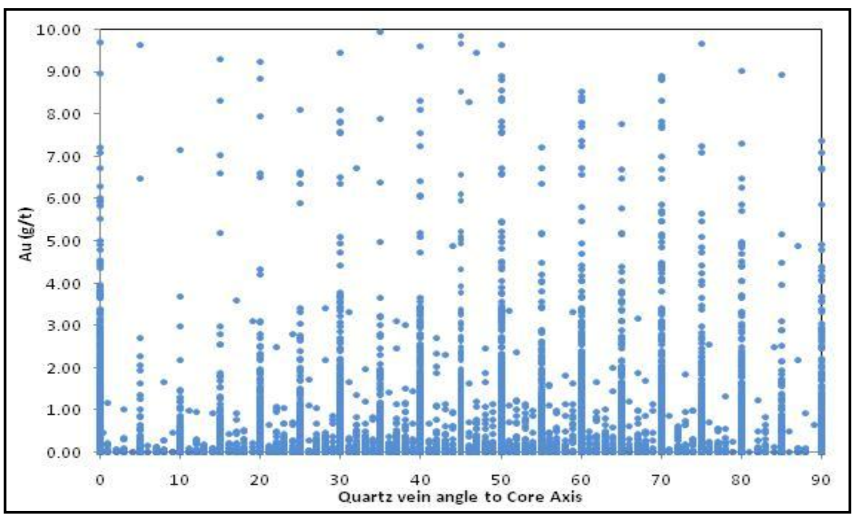
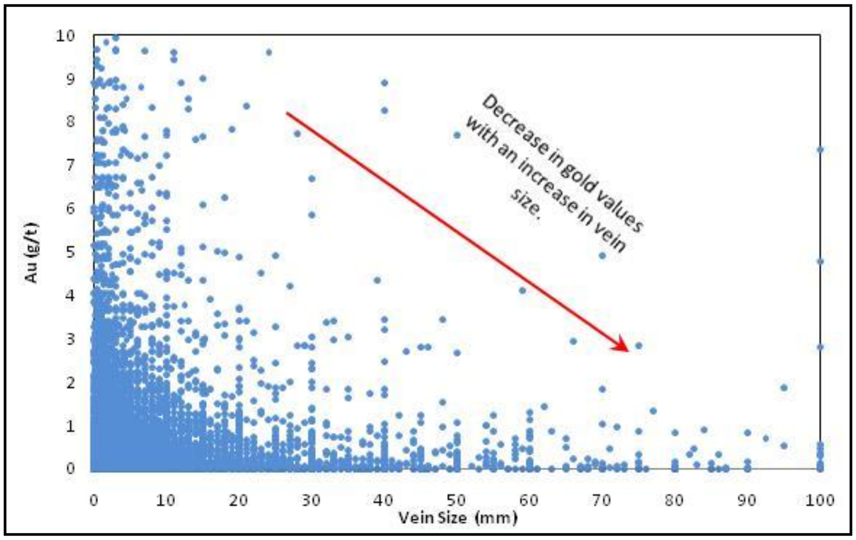


Figure 9-4: Sample Grade of Gold vs. Quartz Vein Size



9.3 Detailed Sampling

In order to better understand gold mineralization, detailed sampling of drill core was undertaken in hopes of determining where the gold resides in the rock.

Singh and the Spanish Mountain field crew completed resampling of selected samples across 20 cm intervals in 2007 and 2008. The intent of this sampling was to determine if gold grades could be approximated by visual estimates of alteration and mineralization, and if detailed sampling could negate the “nugget effect” observed in the gold mineralization. Table 9-2 summarizes the results from this resampling.

When comparing gold averages over the detailed sample interval with the original assay value, several of the detailed samples grades showed a decrease in grade. Analysis of the coarse and fine fraction from the metallic screen data suggest that these samples do not have a pronounced nugget affect and therefore were not suitable candidates for this type of sampling. Further detailed sampling needs to be completed on samples where a clear nugget affect is observed in the metallic screen data to determine if this type of sampling reduces the “nugget affect.”

Table 9-1: Summary of Results of Detailed Core Sampling

Orig. Sample Number	Drill Hole Number	From (m)	To (m)	Original Sample (g/t Au)	Detailed Samples (g/t Au)
E82086	05-DDH-252	112.50	114.00	2.63	3.33
E155034	06-DDH-501	46.00	47.50	1.70	1.66
E155035	06-DDH-501	47.50	49.00	4.46	1.82
E155037	06-DDH-501	49.00	50.50	1.61	1.21
E155256	06-DDH-502	82.00	83.50	1.22	0.71
E155258	06-DDH-502	83.50	85.00	2.67	0.78
E155301	06-DDH-502	142.00	143.50	1.64	1.15
G29172	06-DDH-563	71.50	73.00	1.65	1.57
S07-04662	07-DDH-664	85.00	86.50	0.88	4.29
S07-20644	08-DDH-706	272.50	274.00	1.02	0.72
S07-22740	08-DDH-741	71.00	72.50	1.11	1.37
S07-22741	08-DDH-741	72.50	74.00	3.38	1.50
S07-23047	08-DDH-744	208.50	210.00	1.57	0.69
S07-23496	08-DDH-749	273.50	275.00	0.98	0.95
S07-30513	07-DDH-604	27.00	29.50	2.44	3.38
S07-30545	07-DDH-604	71.50	73.00	0.48	0.08
S07-30547	07-DDH-604	74.50	76.00	0.62	0.91
S07-30569	07-DDH-604	104.50	106.00	0.61	0.61
S07-30509	06-DDH-541	20.00	21.50	0.51	0.35
S07-30550	06-DDH-541	79.00	80.50	0.37	0.20
			Average	**1.58**	**1.36**

9.4 Detailed Analysis of Gold Grades

In April 2007, gravity testwork was carried out on 13 samples selected from coarse reject material of two Spanish Mountain diamond drill holes. The purpose of this study was to compare results obtained by metallic screen analysis with the assumption that fine gold particles are not being captured by the metallic screen process, and to better understand the distribution of gold mineralization. The samples were shipped to the Knelson Research & Technology Centre (Knelson) laboratory in Langley, BC, from Eco-Tech in Kamloops. Samples were composited by Singh into batches with a minimum weight of 10 kg. Each sample was then processed by Knelson using a Knelson Gravity Grade Test (Tran, 2008); pan concentrates and tails were assayed by International Plasma Labs Limited (iPL) in Richmond, BC. iPL is an ISO-certified laboratory.

The samples were ground in a laboratory rod mill to liberate gold within the sample and then processed using Knelson bench scale enhanced gravity centrifuge (Tran, 2008) to recover free gold particles. The concentrate from the test was assayed to extinction and added to the average of two fire assays from the tailings. This test was designed to best mimic the metallic screen process used by Eco-Tech with the screen process replaced by the Knelson gravity separation process.

The two drill holes (699 and 702) were selected based on their grade and geographic location. An overall increase of 21.8% was observed in gold grade. Initial assumptions attributed this increase to fine gold particles, which are not captured in the metallic screen process. However, upon detailed analysis of the fine fractions assays, it was apparent that the increase in gold grades is more affected by the erratic nature of the assay of the fine fraction. Table 9-2 shows the detailed data from this work.

These results imply that there is a significant amount of gold (>40%) that is not separated by the Knelson concentrator. Given that, the total weight of the concentrate was generally less than 1% (Tran, 2008,) the testwork would indicate that not all of the pyrite was accounted for in the gravity separation. In fact, both the screen metallic process and the Knelson gravity process produce the same erratic results in the fine fraction assay; this would suggest that there is a either a fine nugget effect which is not captured by metallic screening or gravity separation, or that the gold is associated with fine pyrite. Metallurgical testing completed for Spanish Mountain Gold in 2007 suggested that the gold is easily liberated from pyrite with fine grinding, suggesting that it may reside in fractures in pyrite and is not chemically bound to pyrite.

A positive correlation exists between the variance of the fine fraction and the variance of the metallic screen assay vs. the Knelson gravity assay. Further work is required to determine how effective the metallic screen process is in those samples where the fine fraction behaves erratically.

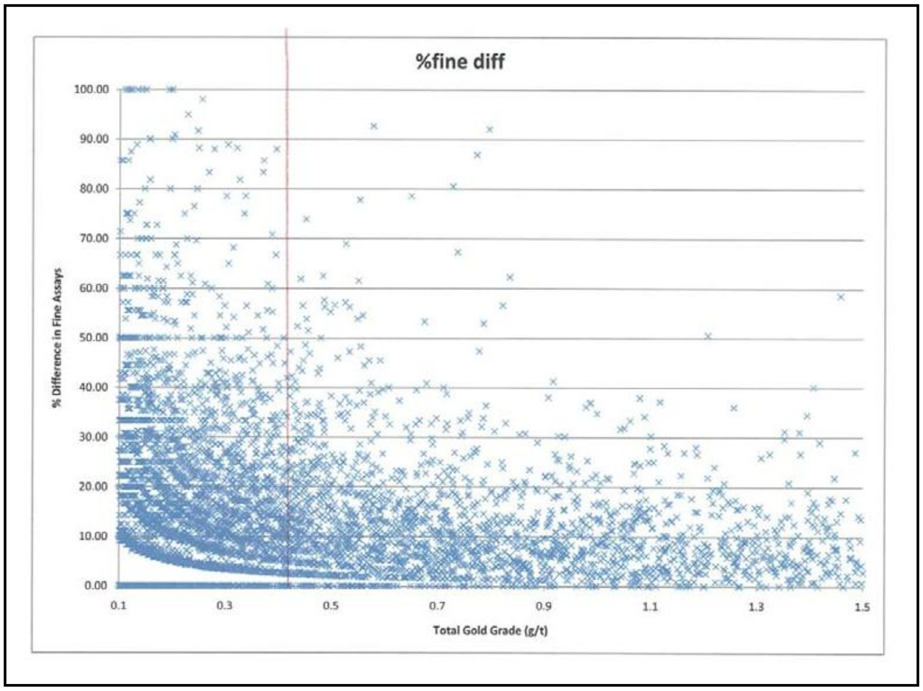
Spanish Mountain Gold has determined that a significant variance exists in the fine fraction in the metallic screen results. Analysis of approximately 6,000 metallic screen samples shows a variance of up to 40% at low grades (>0.45 g/t Au). This variance is particularly significant in low-grade deposits; see Figure 9-5.

The above studies prompted Spanish Mountain Gold to attempt to explain the erratic nature of the fine samples by utilizing a finer screen in the metallic screen process. Eco-Tech was instructed to select a set of 25 samples where coarse gold was noted and to re-screen the fine portion of the sample to extinction utilizing a -200-mesh screen. Results from this work proved inconclusive, as the results did not demonstrate a marked increase in gold grade.

Table 9-2: Comparison of Results of Screen Metallic vs. Gravity Concentration

				Knelson Lab Work							
	Eco-Tech			Au (g/t)				Variance	Gravity	Variance	Variance
Sample	Width	Au (g/t)		Head	Gravity	Fine 1	Fine 2	Fine	Recovery	Head	Screen
K-0001	7.50	0.00		0.05	0.07	0.04	0.04	0.00	46.70	28.57	100.00
K-0002	10.50	0.09		0.08	0.09	0.04	0.03	33.33	62.60	11.11	1.11`
K-0002	10.50	0.09		0.05	0.11	0.04	0.05	20.00	64.60	54.55	19.09
K-0003	6.00	0.09		0.18	2.19	5.10	2.03	151.23	4.90	91.78	96.12
K-0004	7.50	0.03		0.78	0.35	0.12	0.48	75.00	14.20	-122.86	90.29
K-0005	11.50	0.27		0.17	0.27	0.13	0.16	18.75	46.20	37.04	0.37
Repeat	11.50	0.27		0.33	0.26	0.11	0.14	21.43	52.20	-26.92	3.46
K-0006	8.00	1.05		1.23	1.31	0.64	1.18	45.76	31.00	6.11	20.08
K-0007	7.50	0.75		0.41	0.50	0.32	0.29	10.34	40.00	18.00	50.00
Repeat	7.50	0.75		0.57	0.76	0.39	0.40	2.50	49.10	25.00	1.32
K-0008	7.50	1.15		1.40	1.76	1.37	1.56	12.18	17.20	20.45	34.89
K-0009	5.50	0.25		0.58	0.41	0.17	0.21	19.05	54.70	-41.46	38.29
K-0010	6.00	2.60		2.46	2.73	1.27	1.27	0.00	43.00	9.89	4.4
K-0011	7.50	0.59		0.67	0.90	0.68	0.59	15.25	29.70	25.56	34.44
K-0012	7.50	1.22		1.18	1.40	0.82	0.97	15.46	36.40	15.71	12.57
K-0013	7.50	1.31		2.28	1.66	1.26	0.80	57.50	38.50	-37-35	21.20
K-0014	9.00	0.31		0.42	0.38	0.17	0.21	19.05	50.00	-10.53	19.74
K-0015	9.00	0.31		0.42	0.38	0.17	0.21	19.05	50.00	-10.53	19.74
Repeat	9.00	0.31		0.28	0.36	0.21	0.17	23.53	48.00	22.22	15.28
K-0015	6.00	0.29		0.37	0.47	0.49	0.18	172.22	30.00	21.28	38.72
K-0016	7.50	0.17		0.17	0.24	0.18	0.16	12.50	30.30	29.17	30.83
K-0017	7.50	0.47		0.38	0.52	0.36	0.38	5.26	30.10	26.92	10.00

Figure 9-5: Variance Diagram for Duplicate Assays in Fine Fraction of Metallic Screen Analysis



9.5 Gold Associated with Faults

Occurrences of fault zone-related gold mineralization have been recognized at Spanish Mountain. These zones occur in all rock units, and it was thought that they might be responsible for increasing gold grades. Fault zones tend to be graphitic in argillite units and clay-sericite altered in wacke units; faults occur as bedding parallel (thrust) and shallow to vertical normal structures. There is some speculation that this fault zone-related gold mineralization may be a key feature in the economics of the Spanish Mountain property. Wilde et al. (2000) indicated that structures are the key to upgrading the gold grades at the Muruntau deposit.

The Spanish Mountain drill database contains down hole locations of 11,733 fault zones described in 387 diamond drill holes. This is an average of approximately 30 faults per hole. Some occur as 5 cm to 10 cm gouge zones and others are in excess of 10 m wide. In order to analyze the gold association with fault zones, a query of the database was produced that would extract all assays that were greater than a specified grade, as well as how many of those assays had one or more faults included in the sample interval. Figure 9-3 shows a summary of the results of this query.

Table 9-3: Number of Fault Zones within Specified Grade Samples

Specified Grade	Total Assays Returned	Assays Returned[1]	Assays with Faults (%)
>0.25 g/t Au	10,486	1,895	18.1
>0.50 g/t Au	5,663	1,015	17.9
>1.00 g/t Au	2,651	441	16.6
>1.50 g/t Au	1,504	268	17.8
>2.00 g/t Au	952	170	17.9
>5.00 g/t Au	243	49	20.2

Note: [1]·Assays returned with one or more faults included in sample Interval.

The percentage of assayed samples containing faults does not increase as the grade increases, so it can only be concluded that higher gold grades do not have an association with fault intervals. However, it may be useful to plot the higher-grade fault zones in 3D space to determine if there are spatial relationships of the faults with the current resource model.

Singh examined several holes with fault zones and high-grade assay values. One hole, 05-DDH-274, was particularly interesting in that three faults were observed in higher-grade sections and assay values decreased in both directions away from the faults. Fault zones were also noted in hole 06-DDH-289 in the lower wacke sequence, where gold values are typically undetectable. This hole intersected a wide zone of anomalous gold values. Although this section has considerable veining, unlike holes on either side, this hole has far more sericitic-clay gouge seams, which are interpreted as fault zones. This seems to go against the findings shown on Figure 9-3; clearly, more study is needed.

9.6 Comment Regarding "True Thickness"

It is important to note that Spanish Mountain is being drilled as a bulk-tonnage target. As such, the concept of "true thickness" in individual drill hole intercepts has essentially no meaning. The size, continuity, and orientation of mineralized zones are determined from the geometry of multiple drill intercepts.

10 EXPLORATION

10.1 General

This report is concerned primarily with the resource estimate at Spanish Mountain, based on results of sampling diamond drill core from programs in 2005 through 2009. As such, any discussion of earlier exploration programs is very brief. For more details, refer to Lustig and Darney (2006) and Singh (2008), both of which reports are available on SEDAR. Spanish Mountain Gold with Pamicon as the general contractor completed most of the work conducted during this period.

10.2 Pre-2005 Programs

The work in the programs before the 2005 drilling campaign is not directly relevant to the resource estimate reported herein. A summary table of exploration efforts completed is presented in Section 6 – History.

10.3 2005 Program

In 2005, Spanish Mountain Gold completed 7,746 m of diamond drilling from 35 holes and 3,376 m of drilling from 30 RC drill holes. This program was designed as a follow-up on the results of the 2004 RC work by Wildrose, to verify the RC results in diamond drill holes, infill in the main zone, and to test exploration targets, mainly the north zone. Many significant intercepts were encountered and at the end of 2005, an overall understanding of the shape of the deposit, and the distribution of gold mineralization in the several different rock types, had begun to emerge.

10.4 2006 and 2007 Programs

The following was summarized from Peatfield et al. (2009).

In 2006, Spanish Mountain Gold completed 21,881 m of diamond drilling in 88 holes, and 5,009 m of RC drilling in 50 holes. In 2007, Spanish Mountain Gold continued with the drilling program and completed 26,993 m of diamond drilling in 126 holes. Drilling focussed largely on increasing the density of data available on the Main Zone to facilitate estimation of resources. Additional holes tested exploration targets. Metallurgical test-work (see Section 16.0) was completed on material derived from diamond drill samples.

Refer to Singh (2008) for details on core re-logging, geological mapping, rock and soil sampling, airborne geophysics and ortho-photography.

10.5 2008 Program

The following was summarized from Peatfield et al. (2009).

In 2008, Spanish Mountain Gold completed 40,450 m of diamond drilling in 161 holes. Drilling focussed largely on the lateral extents of the main zone specifically to the northwest and to the north at depth.

Drilling also tested exploration targets ROG and CCR. Two significant intersections from the ROG drilling were recovered; however, the controls on mineralization remained poorly understood.

Refer to Peatfield et al. (2009) for details on geological mapping, soil, and rock sampling.

10.6 2009 Program

The following was summarized from Montgomery (2009).

In 2009, Spanish Mountain Gold completed diamond drilling, geological mapping, rock sampling, and preliminary reinterpretation of historic data.

Drilling occurred from July to November and included 62 holes totalling 13,769 m. The drill program included three objectives:

1. Test whether or not the NQ drilling under represented gold grades of the Main Zone. A total of 33 HQ holes totalling 4,671 m and 4 twinned NQ holes, totalling 393 m were completed. The results are discussed in section 12. A comparison of results between the HQ and NQ assay grades show an apparent bias towards higher grade NQ samples. At this time it is unclear as to whether this bias is real as the HQ core samples were analysed at a different lab than the NQ samples

2. Test exploration targets outside of the Main Zone (ROG, Cedar Creek, Placer, North Zone step-out and Black Bear Mountain), with the objectives of finding additional mineral resources, and to gain a better understanding of the geological controls on mineralization. Twenty-one step-out NQ holes totalling 6,849 m were completed. The drilling intersected new areas of mineralization on the property, including northwest of the North Zone resource, where mineralized intersections range up to 63.0 m at 0.75 g/t Au including 28.5 m at 1.0 g/t Au. Anomalous gold values were also intersected in drill holes to the southeast and north of the North Zone and in the Placer area, immediately

west of the Main Zone. Drilling in the southern and south-western portion of the claims, where a volcanic-intrusive domain was delineated, failed to return significant mineralization.

In conjunction with this exploratory drilling, reconnaissance geological mapping and review of previous drill data were carried out in each area drilled. During mapping, 41 rock samples from outcrop were collected to the north on Black Bear Mountain. To the south, in the ROG area, 121 soil samples were collected prior to drilling, extending the extent of soil coverage in this area.

3. Test for mineralization below the Main Zone resource with a series of deep diamond drill holes. To this end, three deep holes totalling 1,705 m were completed in the Main Zone, collared about 200 m apart along a 119° to 289° fence. The drill holes intersected thick successions of sedimentary strata with generally low gold values at depth. Major faults encountered in drilling may represent feeder structures to known mineralization.

   Geological mapping and related work carried out on the property in 2009 resulted in the recognition of NE-trending steep structures believed to control mineralization on the property. Fe-Mg-carbonate alteration forms a 5 km to 8 km halo around the Main Zone resource. Both criteria may be useful during future exploration along the belt.

   Spanish Mountain Gold also completed detailed mapping of the Imperial Metals pit and neighbouring trenches.

11 DRILLING

Drilling in the area of the Spanish Mountain deposit dates from 1947 as summarized below in Table 11-1. Drilling is continuing on the property, with the current resource estimate based entirely on the results from diamond drilling during the period 2005 to the end of 2009. A listing of drill holes used in the resource estimate is included in the Appendix A.

Table 11-1: Summary of Drilling Activity on the Spanish Mountain Property

Year	Company	Drill Type	No. of Holes	Metres	Core size
2009	Spanish Mountain Gold	diamond	62	13,769	NQ & HQ
2008	Spanish Mountain Gold	diamond	161	40,449	NQ & NQ2
2007	Spanish Mountain Gold	diamond	126	26,993	NQ
2006	Spanish Mountain Gold	diamond	88	21,881	NQ
2006	Spanish Mountain Gold	RC	50	5,009	n/a
2005	Spanish Mountain Gold	diamond	35	7,746	NQ
2005	Spanish Mountain Gold	RC	30	3,377	n/a
2004	Wildrose Resources Ltd.	RC	34	2,506	n/a
1999–2000	Imperial Metals Ltd.	blast hole	464	2,542	n/a
1987–1988	Pundata Gold Corporation	diamond	37	3,273	?
1987–1988	Pundata Gold Corporation	RC	15	1,237	n/a
1987–1988	Pundata Gold Corporation	diamond	3	267	HQ
1987–1988	Pundata Gold Corporation	diamond	2	157	NQ
1987	Placer Dome Inc.	RC	7	338	n/a
1986	Mandusa Resources Ltd.	RC	15	833	n/a
1985a	Mt. Calvery Resources Ltd.	RC	8	655	n/a
1985b	Mt. Calvery Resources Ltd.	RC	29	2,521	n/a
1985c	Mt. Calvery Resources Ltd.	diamond	7	?	?
1984	Mt. Calvery Resources Ltd.	diamond	10	467	?
1984	Mt. Calvery Resources Ltd.	RC	10	589	n/a
1947	El Toro BC Mines	diamond	8	792	?

Diamond drilling at Spanish Mountain was for the most part straightforward. LDS Diamond Drilling of Kamloops, BC, and Northstar Drilling of Kelowna, BC, were contracted to provide NQ and NQ2 core, and both have performed exceptionally since commencing work on the project. Average shift production was approximately 68.5 m, with very minimal core loss. Few serious difficulties were encountered, and only a very small number of holes were lost due to bad drilling conditions, generally in fault zones. Core recovery was generally good to excellent.

Collar locations were surveyed in UTM Zone 10N utilizing NAD83 Datum. Final survey work for all drill holes was completed by Crowfoot Surveys of Kamloops, BC, utilizing a transit and Allnorth Consultants of Prince George, BC, utilizing post-corrected differential GPS. Approximately 90% of the diamond drill holes were surveyed by the above methods.

Holes were surveyed at 60 m intervals, using an electronic down hole survey device – the Reflex EZ-shot® instrument. Drill hole deviation was minimal.

An EZ-Mark down hole core-orientation tool was used in 2007 and 2008 for selected drill holes. Orientated data was collected for 194 bedding measurements, 824 quartz vein measurements, and 19 fault zone measurements. Analyses of these data sets correspond with surface structural data and do not contribute any new information. Stereonet plots of bedding measurements collected in the Main and ROG zones agree with their surface data counterparts.

Drilling to date has identified two main styles of gold mineralization which are described in more detail in Section 9. Gold is found within bedded sediments and, to a lesser degree, in quartz veins.

Drilling has identified mineralization at Spanish Mountain in an area that extends approximately 1,500 m x 800 m. From the drill hole data, elevated assay results were observed to be continuous laterally across multiple drill holes and parallel to bedding. These zones range in thickness from 5 m to 150 m; however, the best defined zone of gold mineralization ranges from 5 m to 60 m thick with an extent of 1,200 x 500 m.

12 SAMPLING METHOD AND APPROACH

The following discussion refers specifically to diamond-drill core-samples collected between 2005 and 2009.

Drilling at Spanish Mountain, within the area where the present resource is located, has been completed at variable spacing. In general, holes are located on sections spaced at 50 m intervals, with angle holes spaced 50 m to 100 m apart, but many recent holes were not drilled on section as interpretations have changed as the project has evolved. The majority of the drill holes were drilled at an inclination of 60 degrees, at an azimuth of approximately 120 degrees. The objective was to sample gold in quartz veins. Since 2008, holes were drilled at 120 and at 180 degrees, as well as at other azimuths, to better sample across the stratigraphy as most of the contained metal is not associated with quartz veins but disseminated in the sediments.

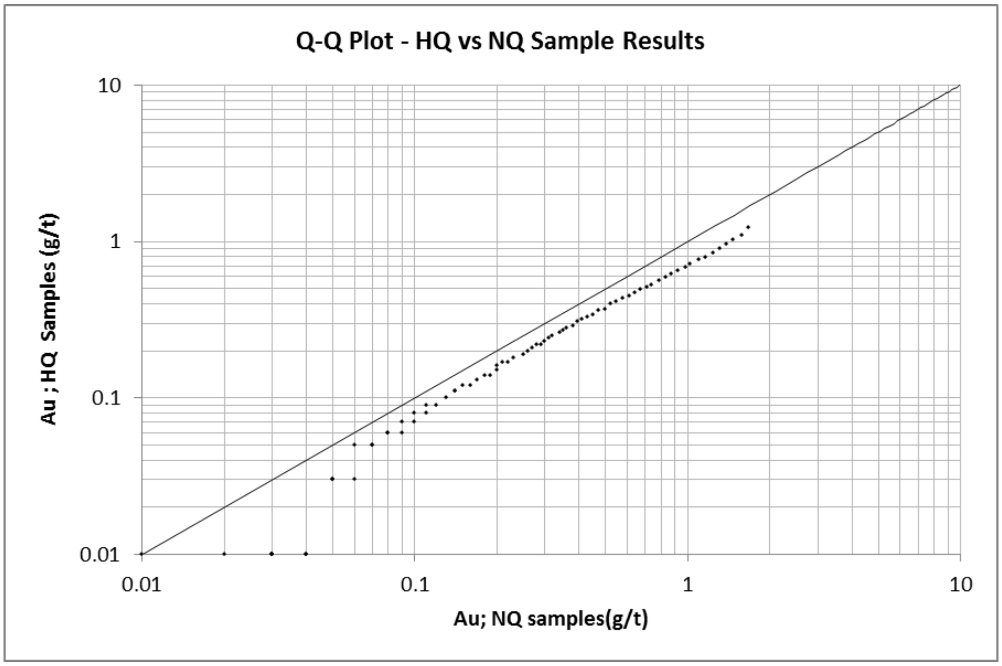
Standard procedure at Spanish Mountain was that NQ or NQ2 size diamond-drill core-recovered at the drill site was transported to a central facility where it was lithologically and geotechnically logged, and sample intervals selected. In most cases, the intervals were 1.5 m long, but some shorter intervals were selected in zones of particular geological interest. The entire length of drill core was sampled for analysis.

A total of 181 samples were collected for bulk specific gravity determination from 40 diamond drill holes completed in 2005 and 2006. The buoyancy procedure was followed whereby the sample mass was determined in air and while suspended in water.

12.1 HQ vs. NQ Test Case

In 2009, a test case was completed to investigate whether NQ sized core samples under stated gold grades. Thirty-three HQ sized diamond drill holes on a 50 m grid were completed to 150 m depth in an area approximately 200 m x 250 m. The results were compared to the assays from NQ samples in the same area to the same depth limit. The comparison was illustrated on a Q-Q plot, which is a scatterplot of two unique datasets with paired results at each percentile from the 10^{th} to the 90^{th} percentile (Figure 12-1). A Q-Q plot mimics a scatterplot typical when comparing duplicate sample results; however, it is used when the pairs come from populations with a different number of samples. In this example, there were 4,759 NQ samples from 57 holes and 2,878 HQ samples from 33 holes. The figure illustrates an apparent bias as the NQ samples consistently returned higher grades than the HQ samples. This cannot be confirmed as the HQ samples were sent to a different lab than the NQ samples for this comparison and the apparent bias may be the result of using different labs. To test whether the apparent bias is real, a selection of pulps from the NQ samples should be sent to ALS for analysis. Assuming however, that the HQ core samples provide a more representative sample than the smaller NQ samples, a sensitivity analysis should be performed whereby the grade of all NQ samples be adjusted and the resource estimate be completed using the same parameters as outlined in Section 17 of this report.

Figure 12-1: Q-Q Plot – HQ vs. NQ Sample Results



13 SAMPLE PREPARATION, ANALYSES, AND SECURITY

The following discussion refers specifically to diamond drill core samples collected between 2005 and 2009.

13.1 Sample Preparation

Company personnel performed all sample preparation prior to shipping the samples to the laboratory.

After logging, drill core samples were cut using a diamond saw, with half of the sample placed in an individual sealed plastic bag, and the second half of the core returned to the core box as a permanent record and stored in racks on the property.

The company conducted a Quality Control (QC) program. Blanks, gold standards, and duplicates were inserted into the sample stream once every 35 samples to provide a check on assay lab data quality. Gold standards were prepared by CDN Resource Laboratories Ltd., of Delta, BC, and certified by Licensed Assayer Duncan Sanderson.

13.2 Sample Analysis

Eco-Tech of Kamloops, BC, an ISO-certified laboratory, completed most of the analytical tests on samples of diamond drill core collected from the Spanish Mountain property until the beginning of the 2009 campaign. ALS Chemex in North Vancouver, BC, also an ISO-certified laboratory, processed most of the samples collected from the 2009 campaign. Both labs followed the procedure as described below.

Gold was determined using a standard 1,000 g screen metallic. The entire half-core sample received was crushed in an oscillating steel jaw crusher. A 1,000 g split was pulverized in a chrome steel ring mill and screened through a 140-mesh screen. Two 30 g splits of the fine fraction (that which passes through the screen) were assayed using a fire assay procedure. The total amount of material remaining above the screen was also assayed by fire assay. The average of the fines assays was weight-averaged with the assay of the coarse fraction to give an overall assay of the sample.

13.3 Sample Security

All sample handling, including core logging and initial sample preparation, has been at secure company-controlled on-site facilities. Prepared samples were placed in sealed woven plastic bags and delivered by contractor personnel or by commercial carriers to analytical laboratories in Kamloops (Eco-Tech) and North Vancouver (ALS).

The author has reviewed the procedures for preparation, analysis, and security of the drill core samples and believes that they were adequate and the results not misleading.

14 DATA VERIFICATION

The following discussion refers specifically to diamond drill core samples collected between 2005 and 2009.

14.1 2005 Drill Program

In each batch of thirty samples, one gold standard, one blank sample, and one duplicate sample was included by Pamicon. Robert Darney, P.Geo., was responsible for monitoring of all assay results, in terms of checking the quality controls, and verifying that the assays in the database matched those on reports received from the laboratories.

14.2 2006 and 2007 Drill Programs

Responsibility for data verification and quality control monitoring was taken over by Singh. The following is a direct quote from a section of his recent report (Singh, 2008):

Gold standards and duplicates were inserted into the sample stream once every 35 samples to provide a check on assay lab data quality. Gold standards were prepared by CDN Resource Laboratories Ltd. of Delta, BC, and certified by Licensed Assayer Duncan Sanderson. Standards in 2006 were initially obtained from WCM Sales Ltd. of Burnaby, BC, but proved problematic, and Spanish Mountain Gold discontinued using them after September 2006.

Sample batches were reanalyzed if any aberrations in the data were observed. In general, the blanks, standards, and duplicates indicate that the assay data are of acceptable quality. There was only one incident where a certificate was rerun. Certificate AK2007-0711 was determined to have highly abnormal potassium values compared to the rest of the data. Upon investigation by the lab, it was determined that the ICP data were contaminated on this certificate, and this certificate was rerun and reissued with corrected data.

14.3 2008 Drill Program

Quality control for the 2008 program remained the responsibility of Spanish Mountain Gold. Peatfield reviewed this work and agreed that the results are generally acceptable. In a very few cases standard assays were outside limits and some re-assaying was requested. In the case of reports from Acme with out-of-limit standards assays, the routine samples all returned very low assays, so no re-assaying was considered necessary. Some re-assaying at ALS did not change the original values appreciably.

14.4 2009 Drill Program

Quality control for the 2009 program remained the responsibility of Spanish Mountain Gold. Mr. Waldegger reviewed QC sample results from samples sent to ALS, and agreed that the results are acceptable.

Standards were submitted with expected gold grades of 0.46, 0.73, 1.16, 2.06, 4.83, and 6.74 g/t Au. Standards performed reasonably well with an overall failure rate of 14%, however failures exceeding 2 standard deviations from the expected value did not occur more than 2 samples in a row. Blanks performed well with a failure rate of 4%. A scatterplot of pulp duplicates samples demonstrated that assay results were repeatable within acceptable limits, returning a correlation coefficient of 0.89

Spanish Mountain Gold did not request any assays to be re-run.

14.5 Independent Assay Database Verification

AGP obtained a 10% selection of assay certificates from each drilling campaign since 2005 directly from the laboratories to compare with the drill hole database provided by the company. There were only a very small number of minor discrepancies, which, taken together, were not serious enough to influence the final estimate.

15 ADJACENT PROPERTIES

Other claim groups that collectively cover large areas of similar geology essentially surround the Spanish Mountain property. Other companies hold surface rights and placer gold operations exist on both Cedar Creek and Spanish Creek.

16 MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 Introduction

Previous testwork on the Spanish Mountain deposit completed in 2007 indicated that gold was associated with pyrite and that in the order of 95% of this gold could be recovered into a flotation concentrate. Alternatives for the recovery of gold, either directly from the ore or from flotation concentrate, clearly indicated that due to active (organic) carbon being present in the deposit it was not possible to use direct cyanidation of either product. It was also established the some of the gold occurred as very fine inclusions in the pyrite and that fine regrinding of the pyrite would be required in order to achieve a high gold extraction. The use of a fine regrind followed by carbon-in-leach cyanidation resulted in a gold extraction in excess of 95% from the concentrate indicating an overall gold recovery of 88 - 90%.

As of 2010, additional grindability, gravity concentration, flotation, and cyanidation testwork has been carried out on three composite samples. The composites used for this new phase of work had head gold grades varying from 0.45 to 0.94 g/t Au and represented different lithologies in the deposit. The testwork is summarized in this section and is detailed in the following reports:

- Spanish Mountain Gold Project – KM2637, Progress Report #1, G&T Metallurgical Services Ltd., 30 August 2010.
- An Investigation into the Grindability of Samples from the Spanish Mountain Deposit, SGS Minerals Services Project #12488-001. (Final Report Pending).

16.2 Sample Description

The current program of gravity/flotation/cyanidation testwork has been conducted on three composite HQ drill core samples derived from hole number 09-DDH-865, as follows:

Composite 865-1..................18.5 – 27.5 m ....Rhyolite Tuff
Composite 865-2..................27.5 – 46 m ....Argillite
Composite 865-3..................55 – 106.5 m ....Rhyolite Tuff

This drill hole, as with the samples used for the 2007 testwork, was located in the starter pit area of the deposit. The head assays for the current composites are based on replicate cuts from the composites, and the averages are summarized in Table 16-1.

Table 16-1: Average Head Analysis Results for the Metallurgical Composites.

Composite	S (%)	C (%)	S_{so4} %	C_{org} %	C_{inorg} %	Au g/t	Ag g/t	Fe %
865-1	1.4	3.27	0.02	0.28	3.03	0.45	1.2	4.81
865-2	2.96	3.22	0.03	1.18	2.04	0.94	1.2	4.12
865-3	1.4	2.3	0.02	0.26	2.05	0.82	0.9	3.32

Composite 2 has a higher gold content than the other samples but also has approximately twice the pyrite content and five times the organic carbon content as the other two composites.

In addition, 24 grindability samples were collected from five different holes in the deposit: 899, 900, 903, 904, and 905.

16.3 Discussion of Results

16.3.1 Grindability

Each of the 24 grindability samples was submitted for bond rod work index (RWi), bond ball mill work index (BWi), and Abrasion index testing. The results are summarized in Table 16-2 and indicate consistent and generally moderate power requirement for grinding of the samples.

Also provided in the table are the results of JK drop weight testing, in particular the Axb and t_a values for nine of the samples in the program. These values, in conjunction with the work index data, were used in the design and the sizing of equipment in the grinding section of the process plant design. The average SG for the samples tested was 2.76.

16.3.2 Rougher Flotation Recovery

All flotation testwork has been conducted at the natural pH, which was consistently in excess of pH 8.2.

Table 16-2: Summary of Grindability Data from Samples of the Spanish Mountain Deposit

Sample Name	Drill Hole	Depth (meter) From	To	Relative Density	JK Parameters A x b[1]	t_a	Work Indices (kWh/t) RWI	BWI	AI (g)
SM-1	899	2.44	40.00		-		12.4	10.9	0.224
SM-2	899	49.38	77.66	2.76	52.5	0.63	13.2	11.9	0.111
SM-3	899	79.85	120.10	2.69	46.7	0.69	14.1	12.9	0.213
SM-4	899	120.10	175.72		-		15.1	13.3	0.251
SM-5	899	175.72	227.00		-		16.1	14.7	0.264
SM-6	900	7.62	52.26				13.8	12.5	0.226
SM-7	900	52.26	116.00		-		15.3	13.5	0.197
SM-8	900	116.00	179.51		-		15.1	13.3	0.215
SM-9	900	186.13	221.28	2.76	53.9	0.61	14.4	13.3	0.215
SM-10	900	243.00	295.53	2.79	47.8	0.92	14.8	13.8	0.298
SM-11	900	306.60	316.72		-		17.5	16.7	0.275
SM-12	900	316.80	328.40		-		16.7	15.7	0.271
SM-13	903	21.00	86.00	2.76	43.0	0.79	13.3	13.0	0.200
SM-14	903	93.50	118.05		-		12.9	13.3	0.173
SM-15	903	127.90	158.35		-		17.0	16.0	0.188
SM-16	903	161.35	237.70	2.80	28.7	0.30	16.3	15.1	0.214
SM-17	903	237.70	270.30		-		16.0	14.5	0.298
SM-18	904	5.18	28.35	2.79	45.8	0.57	12.7	12.6	0.281
SM-19	904	80.00	97.85	2.77	83.2	1.01	12.4	12.7	0.282
SM-20	904	108.40	160.50				13.8	13.7	0.299
SM-21	905	3.36	37.14		-		13.8	12.9	0.207
SM-22	905	37.14	68.58				13.9	12.1	0.120
SM-23	905	85.25	145.80	2.74	51.3	0.62	14.1	15.9	0.281
SM-24	905	149.67	234.25		-		14.2	15.3	0.259
Average:				**2.76**	**50.3**	**0.68**	**14.5**	**13.7**	**0.232**
Stand. Dev.:				0.03	14.4	0.21	1.5	1.5	0.052
Coefficient of Variation (%):				1	29	30	10	11	23
Minimum:				2.69	83.2	1.0	12.4	10.9	0.111
10th Percentile:				2.73	59.8	0.9	12.8	12.2	0.177
25th Percentile:				2.76	52.5	0.8	13.7	12.8	0.206
Median:				**2.76**	**47.8**	**0.6**	**14.2**	**13.3**	**0.225**
75th Percentile:				2.79	45.8	0.6	15.5	14.8	0.277
90th Percentile:				2.79	40.1	0.5	16.6	15.9	0.293
Maximum:				2.80	28.7	0.3	17.5	16.7	0.299

Flotation testwork was initiated on Composite 3 as the most material was available for this composite. The initial tests all utilized 10 kg of feed per test and included grinding to various degrees of fineness followed by gravity concentration by means of a laboratory Knelson concentrator and staged sulphide flotation. The Knelson concentrate was further upgraded by hand panning to give an indication of how much of the gold was recoverable by pure gravity means. The pan concentrate and tailing were assayed separately from the flotation products. The object was to evaluate if an improvement in recovery might be realized if the Knelson product were to bypass the flotation stage and go directly to regrinding, ahead of cyanidation. Gravity concentration is discussed in greater detail later in this section.

It is apparent in Figure 16-1 that across the range of grinds from 80% passing 78 to 184 µm, there is no discernable pattern of grind sensitivity while at 272 µm there appears to be a loss in recovery. Following this initial series of tests a grind size of 184 µm was selected as the target size and was used for all subsequent tests on this and the other composites. Figure 16-2 summarizes the average flotation tailings assays from all nine tests conducted on Composite 865-3.

The results in Figure 16-2 indicate that the variation in the tailings assays at a given grind size can be comparable to the variation across a wide range of grind sizes. The variations at a given grind size result from differences in collector addition and flotation time. At the 184 micron size for instance, the maximum tailings grade (test 12) was obtained with 6 minutes flotation and 35 g/t collector addition while the minimum tails grade (test 15) was obtained with 8 minutes flotation time and 60 g/t collector addition. Considering that other samples have greater sulphide and carbon content, the higher collector additions and longer float times are mandatory. Although for this composite a rougher flotation time of 6 minutes appeared adequate when increased collector additions were employed, in order to allow for variations in composition, a batch rougher flotation time of at least 8 minutes is recommended. Additional testwork is required to optimize the collector addition. As only a single point is available at the coarsest grind tested, additional tests with increased collector addition should be conducted during the Pre-Feasibility Study on several composites at coarser grinds to determine if a coarser grind than 184 µm is economically justified.

The metallurgical balance for each test is based on the average of four assays for the rougher flotation tailings as some variation was noted in the assays for a given tailing sample even with better than 90% Au recovery as can be seen in the compilation of replicate tails assays in Table 16-3. The variation in tailings assays is not excessive for a low-grade gold deposit of this nature.

Tests were carried out on Composite 3 at a grind of 184 µm with and without gravity concentration to establish if the gravity step was having an effect on the overall recovery. The results in Figure 16-3, and supported by comparison of the tailings assays for tests 3 and 5 in Table 16-3, indicate that the gravity concentration stage may have increased the recovery slightly, although gravity concentration has not been optimized. While the increase in recovery may not appear to be sufficient to warrant the cost of the gravity concentration stage, there are other considerations that may justify the inclusion of this circuit.

Figure 16-1: Effect of Grind Size on Rougher Flotation

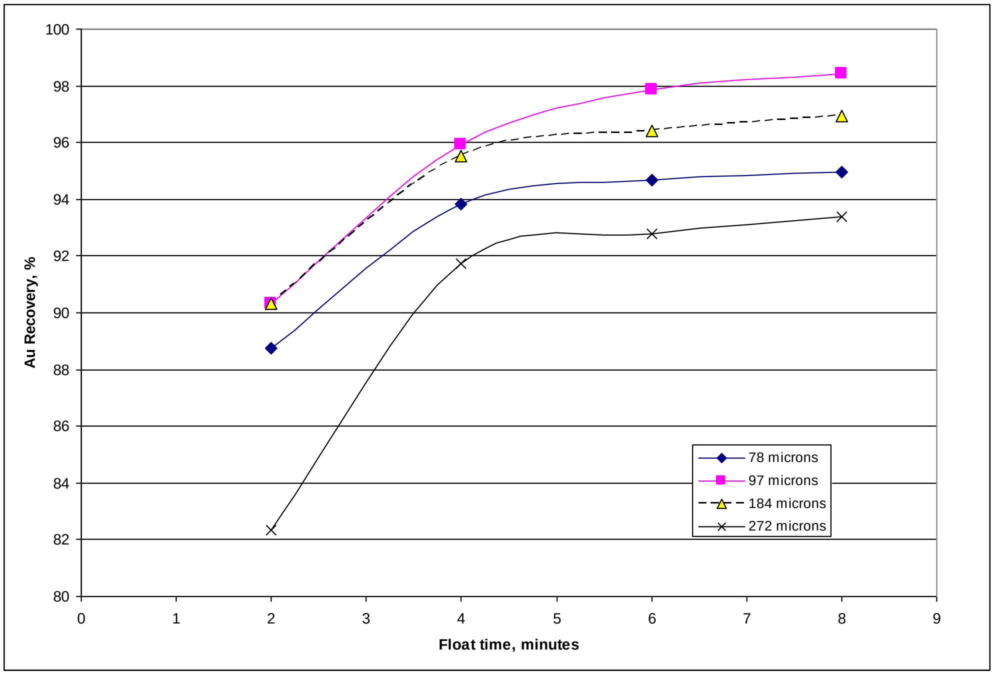


Figure 16-2: Rougher Flotation Tailings Assays as a Function of Grind Size

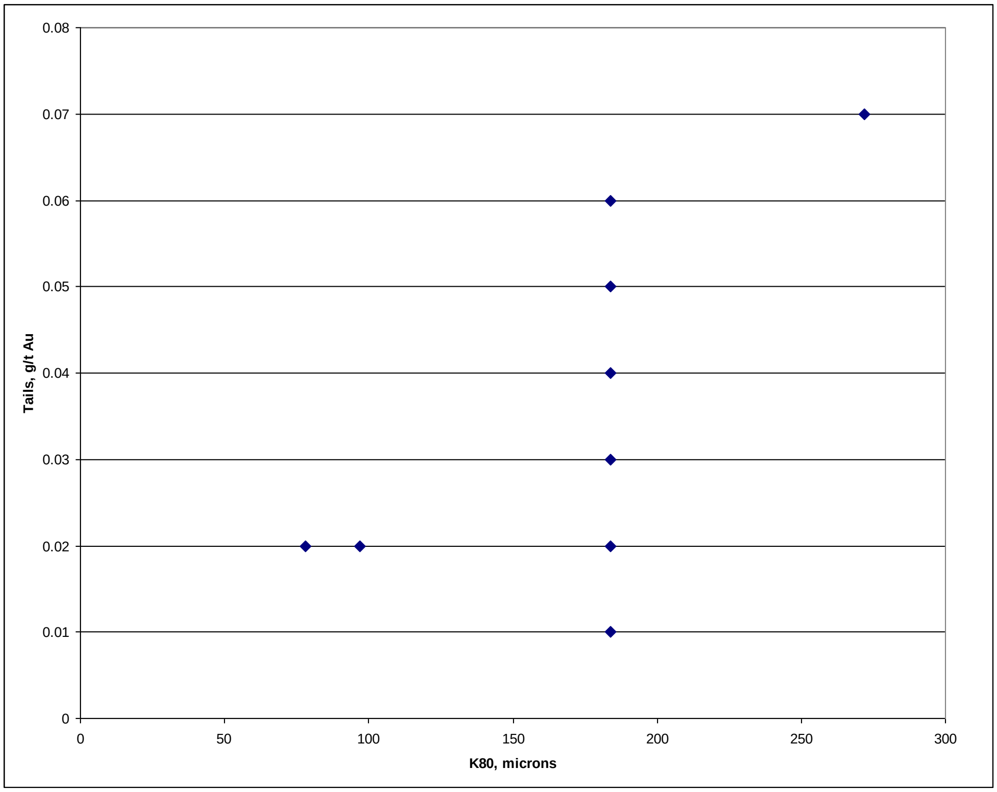


Table 16-3: Replicate Tailings Assays

Test No.	Grind P_{80}	Cut 1	Cut 2	Cut 3	Cut 4	Avg.
1	78	0.04	0.01	0.03	0.01	0.02
2	97	0.01	0.01	0.03	0.03	0.02
3	184	0.03	0.03	0.03	0.03	0.03
4	272	0.06	0.07	0.07	0.06	0.04
5	184	0.02	0.04	0.05	0.05	0.05
10	184	0.06	0.04	0.03	0.05	0.05
12	184	0.06	0.06	0.08	0.06	0.06
15	184	0.02	0.02	0.01	0.01	0.01
18	184	0.02	0.02	0.02	0.02	0.02

Figure 16-3: Effect of Gravity Concentration on Overall Recovery

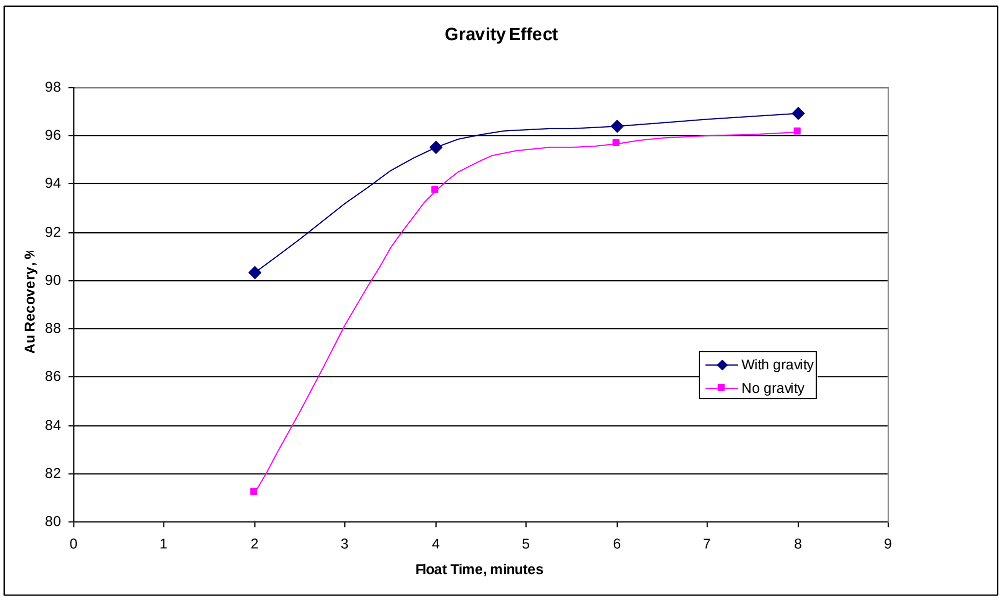


Table 16-4 summarizes the results for tests carried out on Composite 3 at a grind of 184 µm. Both the rougher weight percent and the total rougher recovery include the gravity and flotation recoveries. It appears that the use of the low collector addition together with a short flotation time in test 12 resulted in a low mass pull and a corresponding low recovery. The average result at the bottom of the table excludes the result of test 12. While 45 g/t PAX appears to be adequate for this composite, in order to ensure adequate recovery during subsequent cleaning, a collector addition of 60 g/t PAX is recommended with a batch rougher flotation time of 8 minutes.

Table 16-4: Rougher Recovery Results for Composite 3

Test No.	Gravity (Yes/No)	Rougher Wt. %	Rougher Recovery (%)	PAX (g/t)	Float Time (min)
3	Y	11.5	94.9	45	8
5	N	8.9	96.2	45	8
10	N	12.4	94.7	60	6
12	Y	8.1	92.0	35	6
15	Y	11.4	98.9	60	8
18	Y	11.1	97.8	60	10
Average		11.1	96.5		

In order to determine if consistent results would be achieved with the other composites, comparative flotation tests were carried out without gravity concentration on the other two samples at a grind P_{80} of about 180 µm to 185 µm. The results are summarized in Figure 16-4.

Composites 1 and 3 gave the same response while Composite 2 resulted in a lower recovery. Composite 2 is from a different rock-type than the other two composites but also has more than double the sulphide content of the other two composites and a higher graphite (organic carbon) content that could also result in increased reagent consumption. The total mass pull to the rougher concentrate for Comp 2 was only 8.8% in spite of the higher sulphide content. Additional tests were carried out on this composite with gravity concentration and increased collector additions.

Over the course of the testwork, the variables introduced during rougher recovery include the introduction of gravity concentration ahead of flotation, collector addition to the roughers and total rougher flotation time. The parameters and results for Composites 1 and 2 are summarized in Table 16-5 and Table 16-6.

Composite 1 behaves very much the same as Composite 3 with good recovery being achieved even with low collector addition and short flotation time. The inclusion of gravity concentration appears to result in a marginally lower tailing assay.

Figure 16-4: Comparative Results for Three Composites

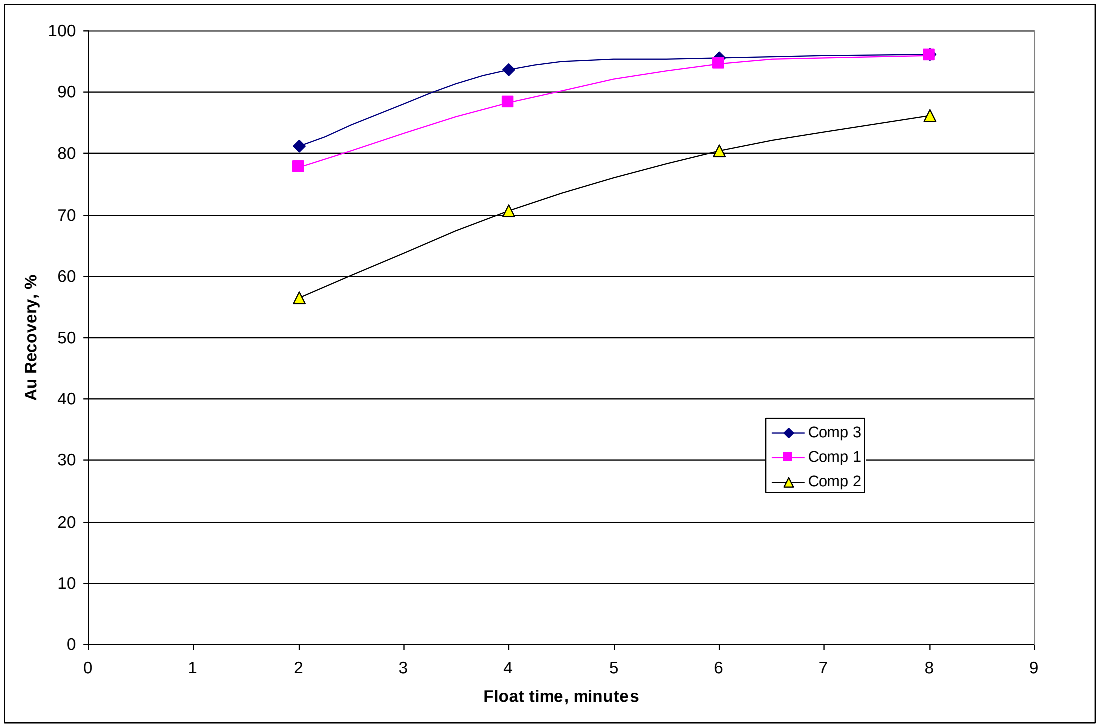


Table 16-5: Summary of Rougher results for Composite 1

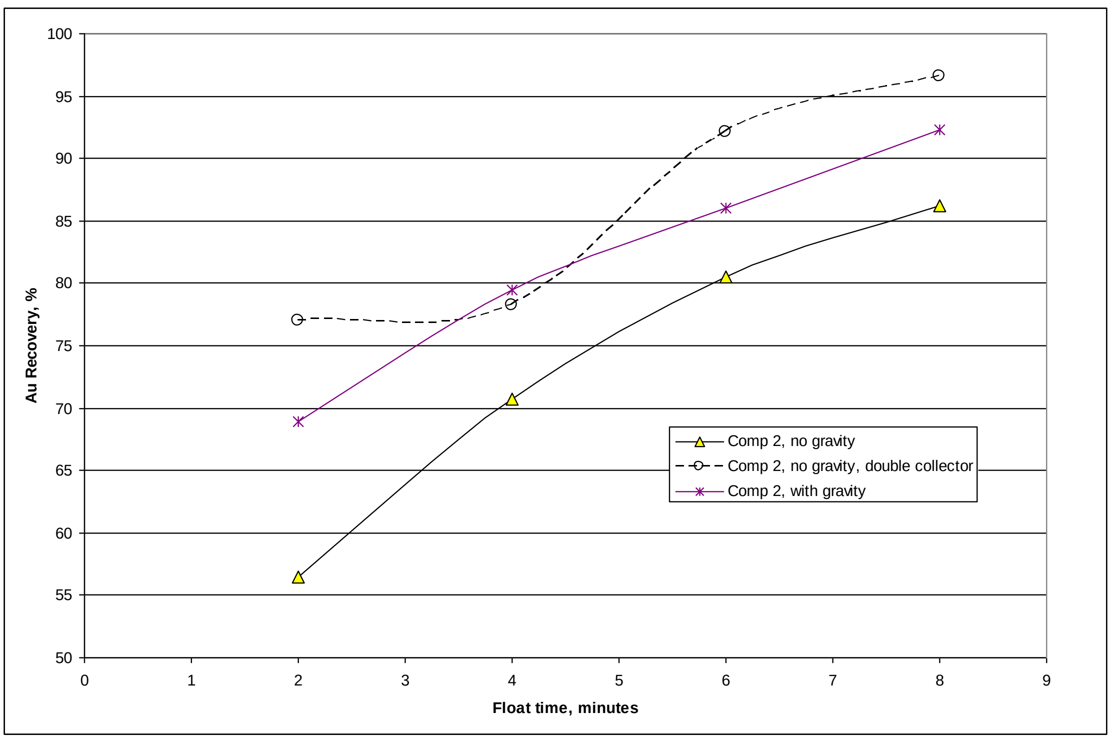
Test No.	Gravity Y/N	PAX (g/t)	Rougher Time (min)	Tailing Au g/t
6	N	45	8	**0.02**
11	Y	60	8	**0.01**
13	Y	35	6	**0.01**
16	**Y**	**60**	**6**	**0.06**

Table 16-6: Summary of Rougher Results for Composite 2

Test No.	Gravity Y/N	PAX (g/t)	Rougher Time (min)	Tailing Au g/t
7	N	45	8	**0.14**
8	N	90	8	**0.04**
9	Y	45	8	**0.09**
14	Y	35	6	**0.22**
17	Y	60	6	**0.05**
25	**Y**	**120**	**12**	**0.04**

Composite 2 gave a high tailing assay with low collector addition, with and without gravity concentration. The tailing assays are higher for this composite than for the others even under optimum conditions due to the higher head assay, although the overall recovery is essentially the same. For this composite, the inclusion of gravity concentration reduces the need for increased collector addition somewhat (test 7 vs. test 9) but a more dramatic improvement in recovery is achieved through increased collector addition (test 8 vs. test 7 and test 17 vs. test 14). The comparison of cumulative recovery with and without gravity and with increased collector for this composite is shown in Figure 16-5. The net conclusion for this composite is that a PAX addition of 90 g/t with a rougher flotation time of 8 minutes is recommended even though the inclusion of gravity concentration may mitigate the benefit of the increased collector addition. While test 25 achieved the same tailing assay as test 8, the total rougher mass pull was 22.9% vs. 16.1% in test 8 so that the additional collector and flotation time used in this test merely floated additional gangue with no benefit to overall recovery.

Figure 16-5: Effect of gravity Concentration and Increased Collector Addition on Recovery Kinetics for Composite 2



The overall rougher results for the three composites are summarized in Table 16-7. The rougher recovery is essentially constant across a wide range of feed gold content while the rougher mass pull increases with the sulphide content of the material.

Table 16-7: Average Rougher Flotation Results

Composite	Test No.	Au g/t	S %	Au Recovery %	Rougher Wt. %
1	11	0.45	1.40	97.9	9.9
2	8	0.94	2.96	96.6	16.1
3	Avg.	0.82	1.40	96.5	11.1

For a feed having a sulphur content of 2%, a rougher mass pull of 13 weight% is projected with a rougher gold recovery of 96.5%

16.3.3 Cleaner Flotation

Testwork has been conducted on cleaner flotation of the rougher concentrates. The overall flowsheet that is proposed is one wherein the gravity concentrate is directed to regrinding and cyanidation while the rougher float concentrate is subjected to a single stage of open circuit cleaning before regrinding. A significant reduction in the mass to be reground and leached is achieved through cleaning with a loss of about 1.5% of the gold. The mass reduction is illustrated by the results summarized in Table 16-8. It is apparent that gold losses to the cleaner tails could be 1% or less as long as sufficient reagent additions are made during both rougher and cleaner flotation. For the purpose of the PEA a gold loss at this stage of 1.5% is recommended and the overall flotation recovery of 95% is therefore supported. Under these conditions a mass reduction of about 4% of the feed weight is indicated.

Table 16-8: Summary of Cleaner Test Results

Test No.	PAX (g/t) Ro/Cl	Ro Conc. (Wt %)	Cl Conc. (Wt %)	Cl Tail Wt Loss (%)	Au Loss to Cleaners (%)
Composite 1					
13	35/10	6.5	2.4	4.1	**21.2**
16	60/30			3.7	**1.2**
Composite 2					
14	35/10	7.1	2.5	4.6	**19.8**
17	60/60			4.5	**4.0**
25	120/50			3.4	**0.2**
Composite 3					
10	60/15	12.4	6.9	5.5	**1.0**
12	35/10	7.2	3.1	4.1	**30.0**
15	60/30	10.3	6.1	4.2	**0.6**
18	**60/30**			**4.8**	**2.1**

The duration of cleaner flotation was varied during the test program but 8 minutes of cleaner retention time appears to be adequate for all samples tested.

As well as gold, the samples tested contain on average about 1 g/t Ag. A silver recovery to the cleaned concentrate of about 50% is indicated by results to date.

16.3.4 Gravity Concentration

Table 16-9 summarizes the gravity recovery results for all three composites. For Composite 3, tests 1, 2, and 4 were done at varying grind size but all other results are at the target P_{80} of 180 to 185 µm. The gravity circuit in all cases consisted of passing 10 kg of ground sample through a Knelson concentrator, which produced in the order of 100 g of concentrate. This Knelson concentrate was then hand panned to produce a final upgraded product weighing just a few grams.

Table 16-9: Summary of Gravity Concentration Results

Test No.	Gold Recovery (%)			Gravity Conc. Weight %
	Pan Conc.	Pan Tails	Total Gravity	
Composite 1				
11	13.1	23.3	36.4	1.1
13	9.1	30.2	39.3	0.9
16	7.5			
Average				
Composite 2				
9	7.5	24.3	31.8	1.2
14	4.6	28.8	33.4	0.9
17	16.2			
25				
Average				
Composite 3				
1	14.7	32.3	47.0	0.9
2	1.5	52.1	53.6	0.9
3	1.3	44.7	46.0	1.2
4	0.7	31.4	32.1	1.2
12	2.8	40.0	42.8	0.9
18	16.5			
Average				

The recovery to the Knelson concentrate in the present tests is approximately one third of the total gold for Composites 1 and 2 and approaching half for Composite 3. These results are comparable to those obtained for a series of gravity tests on HQ drill core during 2009. In that program 10 to 12 kg core intervals were ground to passing 100 µm, passed through a falcon concentrator several times to maximize sulphide recovery and then hand panned. The concentrate produced from the Falcon concentrator averaged in the order of 200 grams, i.e. approximately 1 to 2% of the feed mass. The gold recovery to the Falcon concentrate from two separate drill holes averaged 40 to 70% of the total gold. Hand panning of this product resulted in a final concentrate having a weight of about 20 to 30 grams and recovering 20 to 30% of the total gold.

For the results in Table 16-9, the variation in gold recovery to the pan concentrate roughly follows the amount of pan concentrate produced. It can safely be concluded that the production of a smeltable gravity concentrate would result in a gold recovery of only a few percent. The recovery of gold to the Knelson concentrate was not optimized in these tests and could be significantly greater as indicated by the results of gravity recoverable gold testwork completed in 2010. That testwork indicated gravity recoverable gold of 57.7% and 65.2% for Composites 2 and 3 respectively with a 1% mass pull.

The justification for a gravity circuit consisting of only a Knelson concentrator and no further upgrading is the removal of gold from the primary grinding circuit and potential benefits to gold recovery from concentrates. The inclusion of the gravity circuit will also simplify the operation of the flotation circuit at times when the feed sulphide content is fluctuating.

Tests in Table 16-9 not showing gold recovery for the pan tails or gravity concentrate weight are the ones where the pan tails were combined with the cleaner float concentrate for cyanidation testwork.

16.4 Design Criteria

Based on the testwork to date, the design criteria presented in Table 16-10 were generated.

The testwork design criteria presented above were used as the basis for the process plant design criteria presented in Appendix B. The preliminary plant design criteria provide all the specific unit operation process detail required for the equipment sizing and selection.

16.5 Process Flowsheet, Equipment List, and Layout Drawings

From the testwork conducted, a flowsheet was developed consisting of primary crushing, SAG and ball mill grinding (with integrated gravity recovery), froth flotation, carbon-in-leach cyanidation, carbon elution, and gold electrowinning. A schematic of the proposed flowsheet is presented in Figure 16-6.

Table 16-10: Summary of Process Design Criteria

Parameter	Units	Design Data
Au head grade	g/t	0.67
S_{total} head grade	%	1.92
C_{total} head grade	%	2.93
Primary grind size, P_{80}	µm	184
Rougher float time (batch)	min	8
Cleaner float time (batch)	min	8
PAX addition to roughers	g/t	90
PAX addition to cleaners	g/t	30
Rougher mass pull (total)	wt %	13
Gravity circuit mass pull	%	1
Mass loss to cleaner tail	wt %	4
Mass to gold recovery	%	9
Gold recovery to roughers	%	96.5
Gold loss to cleaner tailing	%	1.5
Overall flotation gold recovery	%	95
Regrind P_{80}	µm	20
CIL residence time	h	24
CIL stage recovery	%	95
Overall gold recovery	%	90

Based on the design criteria and the process flowsheet, an equipment list was developed for the 25,000 t/d base-case processing plant. Major process equipment is summarized in Table 16-11. The complete equipment list is provided in Appendix B. The list provides a basis for the processing plant capital cost estimate presented in Section 24 of this report.

A set of layout drawings were completed for the processing plant area that consisted of elevations and plan views. The drawings indicate the general arrangement of equipment in the plant and are essential for estimating the material quantities required for the capital cost calculation. A complete set of the layout drawings for the plant are presented in Appendix B.

Figure 16-6: Spanish Mountain Process Plant Flowsheet

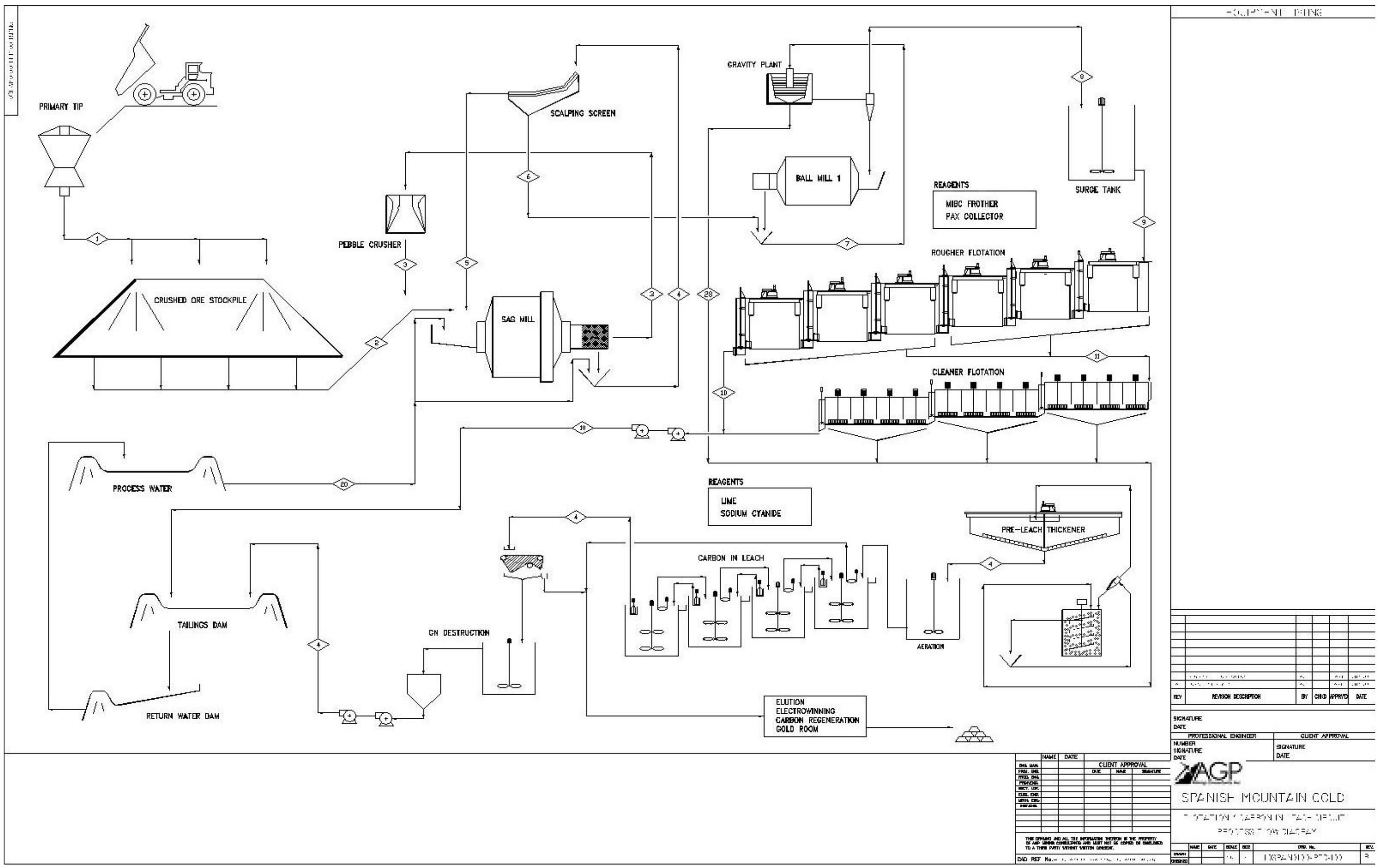


Table 16-11: Spanish Mountain Project Process Plant Equipment List (major equipment only)

Area	Item	No	Type	Description	Specification	kW
100	CGA	04	M	Primary Crusher	Supplier: Metso Type: Superior 42-65	350
100	FCV	32	CON	Stockpile Feed Conveyor	Supplier: Continental Type:	225
100	FCV	52	CON	SAG Mill Feed Conveyor	Supplier: Continental Type:	110
200	MSA	16	M	SAG mill	Supplier: Outotec Type:	5700
200	CCB	46	M	Pebble Crusher	Supplier: Metso Type: HP200	132
200	PCB	58	M	SAG Mill discharge Pump	Supplier: Metso Type: XM400	150
200	SVE	64	M	Scalping Screen	Supplier: Metso Type: 12' x 20' SD	55
200	XLC	78	M	Mill Area Overhead Crane	Supplier: Type: 20t capacity	30
200	MZA	82	M	SAG Mill Liner Handler	Supplier: RME Type: 7-Axis	11
220	MBA	02	M	Ball Mill	Supplier: Outotec Type:	7500
220	PCA	08	M	Ball Mill Discharge Pump A	Supplier: Metso Type: XR 350	300
220	PCA	09	M	Ball Mill Discharge Pump B	Supplier: Metso Type: XR 350	300
220	YAA	12	M	Ball Mill Cyclone Cluster	Supplier: Linatex Size: 10 x 840 mm	-
220	GDA	18	M	Centrifugal Concentrator	Supplier: Knelson Type: XD-70	150
300	XCB	02	M	Rougher Flot Cell #1	Supplier: Metso Type: RCS 160	160
300	PCA	20	M	Rougher Conc Pump A	Supplier: Metso Type: VF 350	92
320	XCB	02	M	1st Cleaner 1 Cell 1	Supplier: Metso Type: RCS 20	37
330	MBB	02	M	Regrind Mill	Supplier: Metso Type: Vertimill 1500WB	1875
330	YAA	08	M	Regrind Cyclone	Supplier: Linatex Size: 6 x 375mm	-
330	PCB	18	M	Regrind Mill Discharge Pump	Supplier: Metso Type: HR250	90
350	ACA	06	M	Concentrate Thickener	Supplier: Westpro Machinery Size: 20 m dia	12
400	TBA	04	P	CIL Tank #1	Supplier: DRAA Type:	-

Area	Item	No	Type	Description	Specification	kW
400	XSA	18	M	CIL Tank #1 Agitator	Supplier: Hayward Gordon Size: LH9	30
420	TCB	14	M	Elution column	Supplier: Summit Valley Eq. Size:	-
420	XGR	46	M	Gold Room (complete)	Supplier: Summit Valley Eq. Size:	125
450	TBA	02	P	CN Destruction Tank #1	Supplier: DRAA Type:	-
450	XSA	06	M	CND #1 Agitator	Supplier: Hayward Gordon Size:	18
450	PPA	12	M	Leach Tailings Discharge Pump	Supplier: Metso Size: HR150	18
500	TBA	14	P	Process Water Tank	Supplier: DRAA Type:	-
500	PCC	16	M	Process Water Pump	Supplier: Metso Type: MM400	150
500	HAC	32	M	Plant Air Compressor	Supplier: Ingersoll Rand Size: GA110	110
500	HBB	50	M	Flotation Air Blower	Supplier: Continental Size: 600-5	110
500	HBB	51	M	Cyanidation Air Blower	Supplier: Continental Size: 600-5	110

16.6 Process Description

16.6.1 Proposed Plant Description

This section describes the parameters used to design a new gold concentrator for the Spanish Mountain Project near Lively, British Columbia, for the base-case operating scenario of 25,000 t/d.

Subsequent economic analysis of the project has indicated an optimal throughput of 40,000 t/d. As a result, the capital and operating costs presented in Section 24 have been scaled from the base-case using the well established scaling method of the six-tenths rule.

The fundamental design criteria have been developed using limited testwork and should be considered preliminary.

16.6.2 Process Summary

The Spanish Mountain concentrator is designed as a nominal 750,000 t/month plant. Mine haul trucks will tip into a primary gyratory crusher station, which is designed for 86%

availability. Surge capacity between the mill and crusher/mine is handled by a ~20,000 tonnes stockpile.

Material is drawn off the stockpile using apron feeders. SAG mill feed control would consist of variable speed feeders plus mill feed size distribution measurement. The SAG mill discharge classification is achieved as follows:

- Trommel screen (40 mm) directs oversize to the pebble crusher via a series of recycle conveyors and allows undersize to gravitate to the SAG mill discharge sump.
- Trommel screen undersize material is further classified by a vibrating, multi-angle scalping screen that cuts at 4 mm, oversize recycling back to the SAG mill, undersize feeding forward to the flash flotation cell.

Scalping screen undersize flows by gravity to the ball milling circuit. The ball mill operates in closed circuit with cyclones. The cyclone pack cuts at a P_{80} of ~184 µm, providing the necessary degree of liberation for good flotation. A 20% split of the cyclone underflow is fed through a gravity concentrator with the concentrate reporting to the regrind mill.

The cyclone overflow reports to the feed box of the rougher flotation circuit. The flotation plant consists of six tank cells for rougher duty. Each cell would have independent pulp level control and air flow control.

The cleaner circuit consists of eight tank cells in series. Cleaner tailings are fed by gravity into the rougher tailings pump box.

Final flotation concentrate is combined with the gravity concentrate and reground in a tower mill to an 80% passing size of 20 µm. The tower mill operates in closed circuit with a cyclone cluster. Cyclone overflow reports to the concentrate thickener.

Reground concentrate is dewatered in a 20 m diameter, high rate thickener. The thickener overflow is pumped through a filter press to recover any slimes value. The thickener underflow, at 55% solids, is pumped to the carbon-in-leach (CIL) circuit.

The CIL circuit consists of seven identical 600 m^3 tanks. The first tank consists of a pH adjustment and pre-aeration stage, whereas the following six tanks are used for cyanide leaching in the presence of activated carbon.

Loaded carbon recovered from the leach circuit is stripped by a ZADRA process circuit, acid washed, and regenerated in a rotary kiln. Gold is recovered through electrowinning, sludge filtering, mercury retort and a melt furnace.

Reagents are stored, mixed, and distributed from a central reagents area. Frother, collector, and promoter are pumped from the reagents area to head tanks in the flotation section from where peristaltic reagent pumps accurately dose to the process. Lime is dosed at 10% strength and thus requires dosing from a ring main. Each ring main take-off is equipped with a flow meter and control valve combination to allow accurate dosing. Cyanide mixing and storage is handled in a separate area with dosing directly to the CIL circuit using peristaltic pumps.

16.6.3 Detailed Process Description

Crushing – Area 100

Ore will be delivered to the primary tip location by 175-tonnes haul trucks at a frequency averaging nine trucks per hour. Peak delivery rate is assumed to be 2,000 dmt/h, equivalent to two trucks dumping simultaneously within a ten-minute period. Ore is discharged directly into the primary crusher throat. This area is served by a 2,000 ft.lb class hydraulic rock breaker (XBA-06) to handle oversize rocks.

The primary (gyratory) crusher can accept 1,000 mm top size and will run with a 200 mm open side setting. Grizzly oversize enters the crusher and discharges by gravity after crushing into a 500-tonnes rail lined surge pocket. An apron feeder is used to withdraw crushed ore from the surge pocket onto a short sacrificial conveyor (FCV-16). This conveyor runs slower, has a lower troughing angle, and is equipped with a self-cleaning belt magnet (EEA-18). FCV-16 discharges through a transfer point (ZAA-24) onto the main stockpile feed conveyor (FCV-32), which in turn feeds up to the crushed ore stockpile.

The crushed ore stockpile provides a live capacity equivalent to ~18 h plant production. Ore is withdrawn from the stockpile via four lined discharge chutes (ZAA-38, 42, 46, 48) and four apron feeders (FCA-40, 41, 44, 45) (two operating, two standby). Each apron feeder is variable speed and capable of providing at least 50% of total mill feed rate. Stockpile discharge chutes are positioned to provide feed with a "coarse" or "fine" tendency. The feeders are automatically controlled to provide optimum mill feed size distribution.

Each apron feeder discharges via a discharge chute onto the SAG mill feed conveyor (FCV-52).

Spillage and run-off in both the primary crusher building and the stockpile feeders tunnel, is pumped (PCE-36, PCE-50) to surface for appropriate handling. An overhead maintenance crane (XLC-56) of 5-tonne capacity serves the primary crusher.

SAG Milling – Area 200

From each stockpile discharge feeder, ore is withdrawn in measured quantities onto the mill feed conveyor (100-FCV-52). This conveyor discharges via head chute (100-ZAA-54) and into the mill feed hopper (ZAA-14). The SAG mill feed material size distribution will be monitored and/or controlled using a high-speed camera system (WIPFRAG or equivalent).

The SAG mill (MAA-16) is a 32 ft diameter x 20 ft long, grate discharge, semi-autogenous grinding mill. Slurry and pebbles exit the mill after passing through the mill discharge grate and pebble ports onto a trommel screen (SRA-20) fixed to the mill discharge trunnion. Trommel screen oversize material (pebbles) is directed by chute (ZAA-22) onto the SAG mill pebble conveyor (FCV-24) for re-crushing. Trommel screen undersize gravitates into the mill discharge sump (TAA-56) where it is further diluted with water. From the discharge sump, coarse slurry is pumped (PCB-58) to the SAG mill scalping screen (SSC-64) via a distributor box (ZAD-62). Screen undersize slurry gravitates via the screen underpan (ZAB-64) through a sampling plant (ZAC-68, XDB-72, XDA-74) to the ball mill circuit. Screen oversize material gravitates via the oversize chute (ZAA-68) back to the SAG mill for re-grinding.

The SAG mill pebble conveyor (FCV-24) transports trommel screen oversize material from the SAG mill to the in-circuit crusher via transfer chutes (ZAA-26, 28) and the pebble crusher feed conveyor (FCV-30). Should the pebble crusher stop, a gate within ZAA-28 automatically directs pebbles into the pebble stockpile area, which is serviced as required by the plant front end loader. Stockpiled pebbles are reclaimed at an appropriate time by re-loading onto FCV-30 via the pebble reclaim bin (ZAA-32). SAG mill scats are removed from the pebble crusher feed by a self-cleaning cross belt magnet (EEA-34) and associated discharge chute (ZAA-36). The tramp metal detector (EEB-40) checks magnet efficiency.

The pebble crusher feed conveyor discharges pebbles via chutes (ZAA-42, 44) into the pebble crusher (CCB-46) where pebbles are crushed to 15 to 20 mm. Crushed pebbles are discharged onto the return conveyor (FCV-48) which transports crushed material back to the SAG mill feed conveyor for re-combination with fresh feed.

SAG mill slurry spillage is collected in a drive-in sump and then returned to process by a submersible slurry pump (PCE-80).

A common overhead crane (XLC-78) serves the milling area (SAG and Ball). Relining is achieved using the common relining machine (MZA-82).

SAG Mill grinding media is stored in a ball bunker (BAB-02) located partway along the mill feed belt. The bunker is served with a small spillage pump (PCE-04) and a ball loading crane and magnet (XLC-08, EEC-10). Balls are added to mill feed at timed intervals via a ball-loading chute (ZAA-06).

Ball Milling – Area 220

After SAG milling, the particle size is further reduced to 80% -184 µm by conventional, closed circuit milling, in a 22 ft diameter x 30 ft long overflow discharge ball mill (MBA-02).

SAG mill scalping screen undersize reports to the ball mill discharge sump (TBA-06), whereupon it is combined with dilution water and ball mill discharge before being pumped (PCB-08,09) to the cyclone classification cluster (YAA-12). The cluster consists of ten 840 mm cyclones, 9 operating and 1 standby.

Cyclone underflow gravitates via piping into a bifurcated chute (ZAA-14) which splits out nominally 25% of the flow for treatment in a centrifugal concentrator (GDA-18). Tailings from the concentrator, as well as the remaining 75% of the flow from the bifurcated chute, report to the feed chute (ZAA-04) of the ball mill. The gravity concentrate is transferred to the regrind mill feed hopper.

Cyclone overflow reports to a linear trash screen (SBA-26) for removal of woodchips and other tramp material prior to flotation. The screened cyclone overflow stream gravitates to the flotation circuit. The stream of woodchips and tramp plastic from the linear screen is dewatered by a woodchip sieve bend (SSA-32) before being dumped in a storage area.

The screened cyclone overflow reports to a sampling station that consists of a sampling launder (ZAC-22) and an automatic sampler (XDB-24).

Spillage contained in the ball mill area is pumped (PCD-38) to the mill discharge sump for re-treatment.

Ball mill grinding media is delivered to the plant in bulk and is stored in the ball mill ball bunker (BAB-48). The ball bunker is serviced by a crawl and electric hoist arrangement (XLC-42) allowing balls to be lifted into a kibble (ZBA-46) using the ball loading magnet (EEC-44) and tipped into the mill via a ball loading chute (ZAA-40).

Rougher Flotation– Area 300

Screened cyclone overflow serves as feed to the rougher flotation section. The rougher bank consists of six 160 m^3 tank cells (XCB-02 to XCB-12) operating in cascade. Flotation air to each cell is supplied by flotation blowers via a low pressure manifold and is flow controlled by modulating valves and vent-captor type flow meters. Pulp level is maintained in each cell by modulating dart valves.

Rougher concentrate is collected in a common launder (ZAC-14) and pumped (PCA-21, 22) to the 1st Cleaner feed box. Rougher tailings from the final rougher tank report to a sampling launder (ZAC-30), primary sampler (XDB-32), and then the rougher tailings tank (TAA-24) where it is pumped (PCA-26) to the tailings dam.

Spillage in the rougher section is collected in a common sump and pumped back into the first rougher cell using a submersible spillage pump (PCD-16).

Cleaner Flotation – Area 320

The 1st Cleaner circuit consists of eight 20 m^3 tank cells in series (XCB-02 to XCB-16). The cells are arranged in cascade. Flotation air is supplied from a low-pressure manifold and is flow controlled by modulating valves and vent-captor type flow meters. Pulp level is maintained in each cell by modulating dart valves.

Cleaner 1 concentrate is collected from all cells in a common launder (ZAC-18) and pumped (PCA-20, PCA-21) to the regrind circuit. Cleaner tails slurry gravitates via a sampling launder (ZAC-34) and sampler (XDB-36) to the rougher tailings tank (300-TAA-24).

The 1st Cleaner area spillage is collected in bunded areas and directed into the cleaner area spillage pump (PCD-32), which pumps back to the 1st Cleaner feed box.

Concentrate Regrind – Area 330

Final concentrate from the 1st Cleaner section is fed to an 1875 kW tower regrind mill (MBB-02) feed inlet. The mill discharge is pumped (PCB-18) to a cluster of six 375 mm cyclones (YAA-08) (5 operating, 1 standby). Cyclone overflow, with a target P_{80} of 20 µm, flows by gravity to the concentrate thickener (350-ACA-06), while the cyclone underflow goes to the mill feed inlet.

A spillage pump (PCE-22) is used to pump the contents of the area sump back into the regrind mill.

Concentrate Thickener – Area 360

Reground concentrate is pumped to the concentrate thickener-sampling box (ZAC-02) and sampler (XDB-04) before entering the concentrate thickener (ACA-06) for dewatering. This thickener is equipped with rake lift, bed level detection, and bed mass monitoring. Thickener overflow gravitates to the spraywater tank for recycling, while the thickener underflow is withdrawn from the cone by a centrifugal underflow pump (PPA-08) and pumped forward to the CIL section, or recycled to the thickener feed if of insufficient density.

Thickener area spillage is recovered by pumping (PCD-12) back to the concentrate thickener.

CIL Cyanidation – Area 400

Thickened final concentrate is pumped to a circuit of seven 600 m^3 (working volume), mechanically agitated, leaching tanks. The first of these tanks (TBA-02, XSA-16) provides a pre-aeration step with pH adjustment to 10.5 by the addition of lime slurry from the lime slurry ring main. Overflow from the pre-aeration tank flows into the first of six CIL tanks.

Carbon-in-leach cyanidation is carried out in six agitated leach tanks (TBA-04 to TBA-14, XSA-18 to XSA-28). The tanks are maintained under constant pH control using lime addition, and cyanide concentration is monitored and target levels achieved through operator controlled peristaltic pumps. Slurry is advanced from one tank to the next using interstage screens (SIA-44 to SIA-54) and carbon is moved counter-currently using carbon transfer pumps (PCG-32 to PCG-40) to minimize attrition.

Final screened discharge from the last tank in the CIL circuit is fed to cyanide destruction. Loaded carbon from CIL tank #1 is pumped to the safety screen feed box (ZAA-60) which feeds the safety screen (SVA-62). The screen oversize is transferred by chute (ZAA-64) to the elution circuit. Safety screen undersize is returned to the CIL circuit.

CIL area spillage is recovered by pumping (PCE-68) back to the pre-aeration tank (TBA-02).

Elution – Area 420

Loaded carbon from the CIL circuit is treated to recover gold and regenerate carbon in the Elution area. Loaded carbon is transferred on a batch basis from a storage tank (TAA-12) to the elution column (TCB-14) to be acid washed and stripped using hot caustic solution (ZADRA process).

The pregnant strip solution is pumped to the electrowinning cell (XWA-42) where gold, silver, and contaminants are recovered as sludge. The sludge is dewatered in a filter press, and transferred to the gold room (XGR-46) for final upgrading to doré bar by mercury retort and melt furnace. The barren solution from the filter press is recycled to the strip solution holding tank.

Barren carbon is removed from the stripping column and fed by rotary valve (FGD-20) to the regeneration kiln (XTB-22).

Spillage in the elution area is pumped (PCG-44) back into the process at the operator's discretion.

Cyanide Destruction – Area 450

CIL tailings from the final leach tank (TBA-14) flow by gravity into the cyanide destruction circuit consisting of two agitated 40 m^3 tanks (TBA-02, 04; XSA-06, 08). The Inco SO_2-air process, using sodium metabisulphite in the presence of a copper sulphate catalyst, achieves cyanide destruction.

Overflow from the second CND tank reports to the tailings discharge sump (TBA-10) and then is pumped (PPA-12) to the tailings dam.

Spillage in the cyanide destruction area is pumped (PCD-18) back into the first CND tank.

Services – Area 500

Overflow water is recovered from the concentrate thickener into the spray water tank (TBA-02) from where it is pumped (PCB-04) via an inline filter (AAH-10) to the process water tank (TBA-14). Filter backwash often carries significant grade and would be piped to the pre-aeration tank for re-processing.

Process water is stored in an insulated tank (TBA-14) and is distributed to the plant by the process water pump (PCC-16). The hose-down water supply pumps (PCC-20) provide plant hosing/flushing water.

The process water tank is also used to feed the diesel powered fire water pump (PEA-24) from a separate (lower) offtake, thus guaranteeing availability.

Clean water is piped into the plant from wells and stored in the plant clean water tank (TBB-26). From the storage tank, water is pumped (PCC-28) around the plant for use as reagent mixing water, slurry pump gland seal water and as required for mill lubrication system cooling.

Two compressors (HAC-32, 34) provide plant and instrument air. A filter (HEA-46) maintains air quality. Instrument air is dried using a refrigeration drier (HAD-42).

Receiver (HFA-48) is provided for compressed and instrument air lines to allow for surges in demand.

Two separate blowers supply low-pressure air to the flotation plant and CIL circuit (HBB-50: flotation; HBB-51: CIL). The blowers are fixed speed, with manifold pressure controlled by a modulating valve on an exhaust line.

Reagents – Area 600

Collector – PAX

Potassium amyl xanthate (PAX) pellets are delivered to site in 1-tonne bags and stored in the reagent storage area. Bags are added to the mixing tank (TDB-04) via the reagent area hoist and collector-loading chute (ZAA-02). Collector is mixed (XSA-06) to 10% solution strength within the tank, then transferred (PCC-10) to the storage tank (TDB-12), ready for distribution. The storage tank capacity and solution strength allow a batch to be mixed every eight hours.

From the storage tank, collector solution is continuously pumped (PCC-14) to the collector head tank (TBB-18) which in turn overflows back to TDB-12. Peristaltic hose pumps (PPA-20, 22, 24, 26) meter collector solution to several addition points throughout the plant.

Reagent spillage is pumped (PCE-28) to the tailings tank for disposal on the tailings dam.

A safety shower (XSH-30) serves the reagent area.

Frother – MIBC

Liquid Methyl Isobutyl Carbinol (MIBC) is delivered to site by tanker then transferred to the storage tank (TDB-32) in the reagent storage area. As delivered (100% strength), frother is pumped to the head tank for dosing by the promoter transfer pump (PCC-34). The head tank (TBB-38) overflows via a return line to the storage tank, providing a continuously circulating flow.

From the head tank, peristaltic hose pumps (PPA-40, 42, 44, 46) meter frother solution to several addition points throughout the plant.

Flocculant – Magnafloc 333

Flocculant powder is delivered to site in 1-tonne bags and stored in the reagent storage area. Bags are lifted by the reagent area crane (XLC-72) and added to the flocculant powder hopper (TAA-48). Powder is withdrawn by the flocculant screw feeder (FJA-50) and blown (HBB-52) through a venturi (XEA-54) to a wetting head (XEA-56) located on top of the mechanically agitated (XSA-58) mixing tank (TDB-60).

From the mixing tank, mixed flocculant can be fed forward (PPA-62) to the storage tanks, or recycled back into the mixing tank to aid mixing. Once mixed, the flocculant should be left for several hours to hydrate.

A storage tank (TDB-66) provides sufficient volume for storage of flocculant whilst the mixed batch hydrates in the mixing tank. From the storage tank, flocculant is pumped (PPA-68) directly to the concentrate thickener.

PH Modifier – Calcium Hydroxide

Lime (calcium hydroxide) is delivered to the plant in bulk and offloaded into the lime hopper (BCA-73) by a vacuum line. Dry lime is metered from the hopper into the agitated mixing tank (TAA-80, XSA-78) by a screw feeder (FJA-75) and mixing plate (ZAA-76). Mixed lime slurry at 10% solids is pumped (PCB-79) to an agitated dosing tank (TAA-82, XSA-74). A circulation pump (PCB-87) supplies lime to the CIL circuit via a ring main.

Gold Lixiviant – Sodium Cyanide

Sodium Cyanide (NaCN) powder is delivered to site in 1-tonne bags and stored in a bunded, secure, storage area. Bags are added to the mixing tank (TDB-90) via the reagent area hoist and loading chute. NaCN is mixed (XSA-92) to 10% solution strength within the tank, then transferred (PCC-100) to the dosing tank (TBB-104), ready for distribution. The mixing tank capacity and solution strength allow a batch to be mixed every six hours.

From the dosing tank, cyanide solution is continuously pumped (PPA-106) through a ring main with several dosing points in the CIL area.

Elution Reagents – NaOH and HCl

Reagents in the elution section consist of 50% strength caustic solution (NaOH) and concentrated hydrochloric acid (HCl). Both are delivered to the plant in 44-gallon drums and metered into the process directly using drum pumps (PPB-108, 110).

Cyanide Destruction Reagents – Sodium Metabisulphite and Copper Sulphate

Powdered sodium metabisulphite ($Na_2S_2O_3$) is delivered to the plant in 25 kg bags, 40 bags to a pallet. The bags are emptied into the agitated mixing tank (TDB-114, XSA-116) via a feed chute (ZAA-112). Mixed metabisulphite solution at 10% concentration is pumped (PCC-124) to a dosing tank (TBB-128). A peristaltic pump (PPA-130) delivers the solution to the cyanide destruction circuit.

Powdered copper sulphate ($CuSO_4$) is delivered to the plant in 25 kg bags, 40 bags to a pallet. The bags are emptied into the agitated mixing tank (TDB-134, XSA-136) via a feed chute (ZAA-132). Mixed copper sulphate solution at 10% concentration is pumped (PCC-144) to a dosing tank (TBB-148). A peristaltic pump (PPA-150) delivers the $CuSO_4$ solution to the cyanide destruction circuit.

17 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

The Spanish Mountain Gold property mineral resources were estimated by Mr. Michael Waldegger, P.Geo., Associate Resource Geologist with AGP.

The following sections describe the resource estimate that was produced with a data cutoff date of 19 November 2010.

17.1 Drill Hole Database

AGP received drill hole data from the Spanish Mountain Gold for 472 diamond drill holes and 114 reverse circulation drill holes which included geological logs and assay results. AGP also received 181 specific gravity results from 40 drill holes.

From this complete data set, AGP used 426 diamond drill holes completed between 2005 and 2009. A list of the drill holes used is presented in Appendix A. Reverse circulation drill hole data completed in 2005 and 2006 were not used. The RC holes because they were drilled in close proximity to were drilled in the same area as the more reliable diamond drill dataset. A list of the drill holes used is presented in Appendix A.

A total of 65,833 sample results were used for the purpose of geological modelling and resource estimation. Descriptive statistics of the sample data are presented in Table 17-1.

Table 17-1: Descriptive Statistics on Drill Hole Sample Data

	Au_ g/a	Length_Int
Valid cases	65,833	65,833
Mean	0.25	1.54
Std. deviation	1.79	0.28
CV	7.14	0.18
Minimum	0	0.5
1st percentile	0	1
5th percentile	0	1.5
25th percentile	0.01	1.5
Median	0.04	1.5
75th percentile	0.16	1.5
95th percentile	0.97	1.97
99th percentile	2.93	2.87
Maximum	241	8.53

17.2 Geological Interpretation

The Spanish Mountain gold deposit was modelled by AGP on section using lines which honoured the drill hole data in 3D. AGP modelled domains based on logged lithology and grade shells (Figure 17-1).

The lithological model represents a simplified version of data presented in the drill logs. Four domains were modelled: overburden, upper argillite, greywacke, and lower argillite.

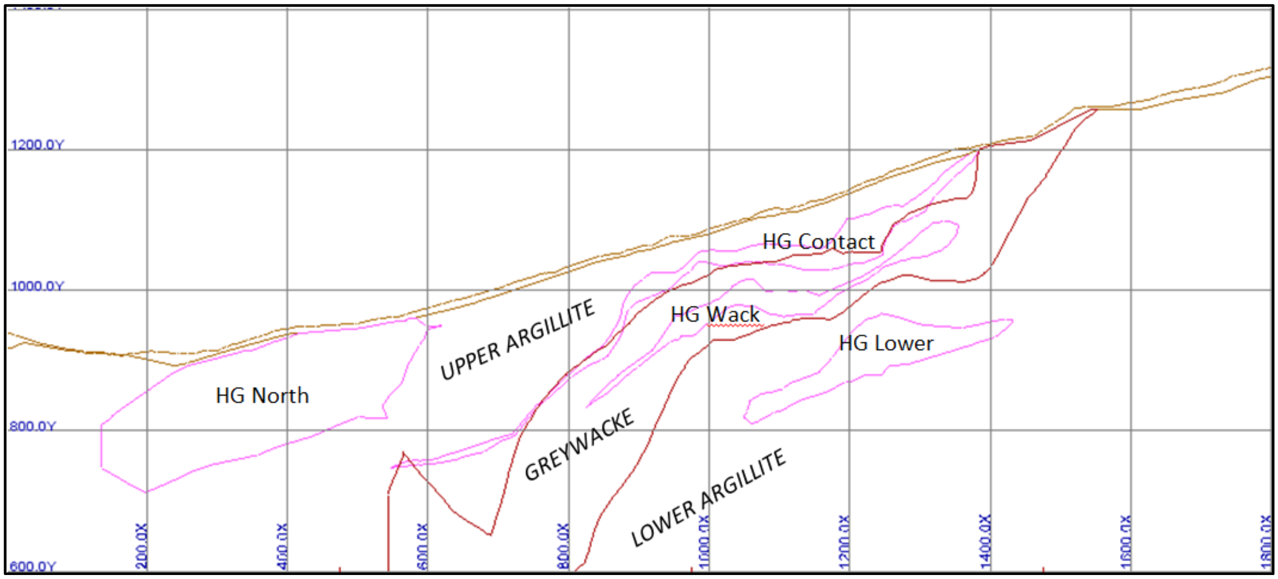
AGP also modelled grade shells based on laterally continuous zones of composited drill hole data greater than 0.6 g/t Au. Visual inspection of the composited data illustrated continuous zones sub-parallel to each other, and one zone, which was coincident with a lithological contact.

Four discrete high-grade domains were modelled:

1. The Upper Contact Domain (hgcontact) is coincident with the hanging wall contact of the greywacke unit and the overlying argillites. Most of the mineralization in this domain is within the argillites; however, the domain straddles the contact. This domain is approximately 30 m thick and ranges in thickness from 5 m to 60 m thick. The domain extends 500 m x 1,200 m.
2. The Greywacke Domain (hgwack) was modelled between the hanging wall and the footwall contacts of the greywacke unit, and is therefore entirely enclosed within this geological domain. This domain is commonly 25 m thick, and ranges in thickness from 5 m to 50 m thick. The domain extends 250 m x 700 m.
3. The Lower Domain (hglower) was modelled within the undifferentiated siltstone and argillite unit, which lies below the greywacke unit. This domain is commonly 60 m thick and ranges in thickness from 20 m to 100 m thick. The domain extends 280 m x 600 m.
4. The North Domain (hgnorth) was modelled within argillite and siltstone to the north of the main deposit. This zone was modelled as a broad domain, which in turn incorporates relatively large intersections of low grade material. Geological and structural control on mineralization is less understood in this area, largely due to a lower density of drilling data available. The domain extends 500 m x 800 m.

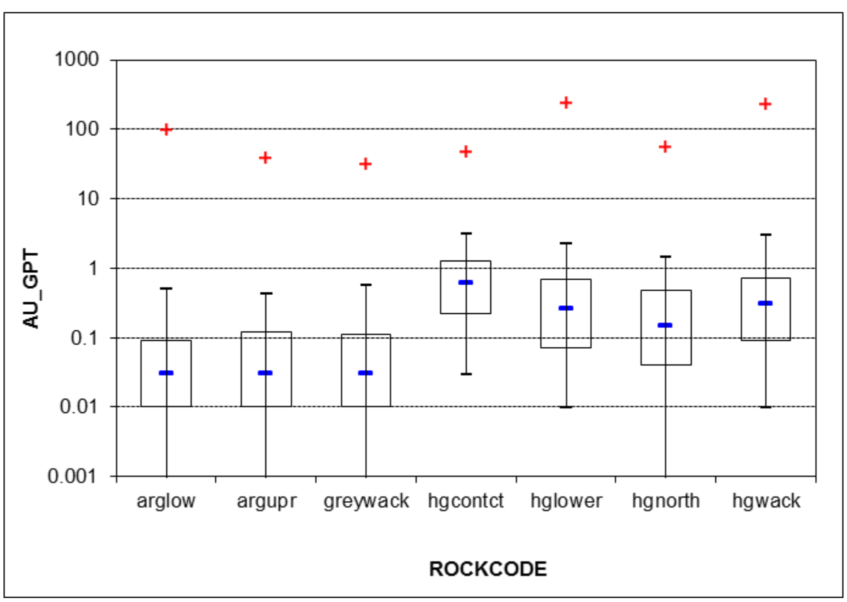
Mineralization occurs outside of the modelled zones; however, at the current drill hole spacing, continuity between relatively thin high-grade intercepts could not be established with reasonable certainty to support the modelling of additional domains.

Figure 17-1: North-South Section Illustrating Lithological Model and Grade Shells



Descriptive statistics of the raw sample data per domain is presented in Appendix A. A graphical representation of the descriptive statistics is illustrated in the box–and-whisker plot in Figure 17-2. In this plot, the interquartile range (25th to 75th percentile) is represented by a rectangle or 'box', and the 'whiskers' extend from the 5th to the 95th percentiles. The red crosses represent the maximum values. The minimum values were all zero and not displayed on this plot.

Figure 17-2: Box and Whisker Plot of Sample Data by Domain

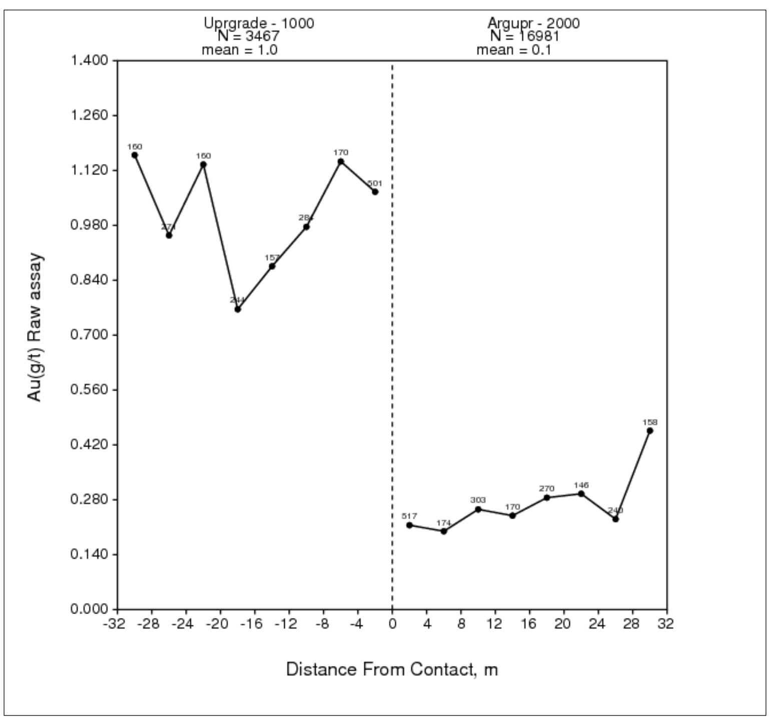


The plot illustrates that the high-grade domains are clearly higher grade than background values for the area sampled by drilling by an approximate order of magnitude or factor of ten. The plot also illustrates that 25% of samples in the low-grade domains are above 0.1 g/t Au and that there are some very high-grade samples in all domains. This is significant in that it shows that there are potential areas of interest within the low-grade domains; however, at the current drill hole spacing it was not possible to group together high-grade values across multiple drill holes to create more high-grade domains.

AGP tested the validity of the high-grade domains using contact plots. The contact plots illustrated that the change in grade across the boundary of the high-grade domains is very sharp, thus supporting the use of domains in resource estimation at Spanish Mountain.

An example of a contact plot is presented in Figure 17-3. The dashed vertical line represents the contact between two domains. The points on either side of the contact line represent the average grade of all points from within the domain found at a distance range from the contact with the neighbouring domain. The plot illustrates that at a separation distance of 0 to 4 m from the contact, the average grade of the 501 samples found within the Upper Contact high-grade domain is approximately 1.1 g/t Au, whereas the 517 samples found within the neighbouring low-grade domain at 0 to 4 m from the contact have an average grade of 0.2 g/t Au.

Figure 17-3: Contact Plot for Upper Contact Domain



17.3 Treatment of High-Grade Outliers

When estimating resources, high-grade outliers can contribute excessively to the total metal content of the estimate. In a geologic context, outliers represent a separate grade population characterized by its own continuity; generally, the physical continuity of high grade is much less than that of the more prevalent low grades. Thus serious overestimation of both tonnage and average grade above a cutoff grade can result if a general model, normally dominated by the lower, more continuous grades, is applied to very high-grade values. The problem is further exaggerated when the high-grade samples are isolated in a field of lower grade samples (Sinclair, 2002).

Decile analyses were conducted and histograms and probability plots were reviewed to determine the potential risk of grade distortion from higher-grade assays. AGP elected to use a two-fold approach:

1. Applied a capping level customized for each domain on the raw assay prior to compositing.
2. Imposed a limited range search restriction on outliers customized for each domain during interpolation of block grade (see Section 17.1.1).

The methodology employed has the benefit of limiting the grade distortion from high-grade outliers by limiting their range of influence to neighbouring blocks only under the presumption that true outliers generally have restricted physical continuity. The high-grade values are acknowledged in the model but their spatial influences are limited.

AGP capped a small number of samples at gold grades outlined in Table 17-2.

Table 17-2: Capping Levels

Domain	Capping Threshold Au (g/t)	Number of Samples Above Threshold	Percentage of Samples Capped
Upper Contact	40	1	0.03%
Greywacke	40	3	0.15%
Lower	30	2	0.12%
North	20	2	0.06%
Upper Argillite	20	2	0.01%
Greywacke	20	6	0.05%
Lower Argillite	30	3	0.01%

Descriptive statistics of the capped sample data per domain are presented in the Appendix A.

17.4 Composites

In order to normalize the assay data, samples are often composited to a standard length. The raw and capped assay values were composited to a 2.5 m length starting at the drill hole collars, honouring the modelled geological boundaries. Any composite less than 1.25 m in length was added to the previous composite, thereby creating a dataset of composites ranging from 1.25 m to 3.25 m in length. The effect of compositing sample data reduced the sample variability with very little negative effect on the mean grade of the sample population. Descriptive statistics of the composited sample data are presented in Appendix A.

17.5 Spatial Analysis of Grades

Geostatisticians use a variety of tools to describe the pattern of spatial continuity or strength of the spatial similarity of a variable. The correlogram measures the correlation between data values as a function of their separation distance and direction. The distance at which the correlogram reaches the maximum variance is called the "range of correlation," or simply the range. The range of the correlogram corresponds roughly to the more qualitative notion of the "range of influence" of a sample; it is the distance over which sample values show some persistence or covariance.

Using Sage 2001 software, variography was completed within the Upper Contact Domain and the surrounding Upper Argilite and the combined Greywacke and Lower Argillite. Reasonable variograms were not successfully completed for the remaining high-grade domains (Greywacke, Lower, and North).

Experimental correlograms were calculated along horizontal azimuths of 0, 30, 60, 90, 120, 150, 180, 210, 240, 270, 300, and 330 degrees. For each azimuth, experimental correlograms were also calculated at dips of 30° and 60°. Lastly, a correlogram was calculated in the vertical direction. Using the 37 correlograms, an algorithm determined the best-fit model. This model is described by the nugget (C_0), up to two nested structure variance contributions (C_1, C_2), ranges for the variance contributions, and the model type (spherical or exponential). After fitting the variance parameters, the algorithm then fits an ellipsoid to the 37 ranges from the directional models for each structure. The final models of anisotropy are presented in Table 17-3.

Table 17-3: Zone 2 Correlogram Models

Domain	Component	Contribution	Rotation Axis Z	Rotation Axis X	Rotation Axis Z	Range X	Range Y	Range Z
Upper Contact	C_0	0.323						
	C_1(shp)	0.537	-73	-24	72	14.1	51.4	4.4
	C_2(sph)	0.14	90	-69	-73	26.1	45.1	86.2
Upper Argillite	C_0	0.383						
	C_1(shp)	0.351	-39	-41	25	51.5	34.8	3.7
	C_2(sph)	0.266	-19	-31	79	125	125	53.3
Combined Greywacke + Lower Argillite	C_0	0.35						
	C_1(exp)	0.65	6	-13	-12	21.6	66.7	8

Note: Rotation angles were set to correspond to Gemcom© Software's rotational convention, which follows the right hand rule with rotation about Z-axis being positive when X moves towards the Y-axis; rotation about the X-axis is positive when Y moves towards the Z-axis.

17.6 Specific Gravity

A total of 181 samples were collected for bulk specific gravity (SG) determination from 40 diamond drill holes. The buoyancy procedure was followed whereby the sample mass was determined in air and while suspended in water. The bulk SG ranged from 2.57 to 3.05 with a mean of 2.78.

17.7 Resource Block Model

The geological interpretation and resource modelling were carried out using 3D geological modelling software provided by Gemcom© Software International Inc. (Gemcom©) of Vancouver. Modelling was carried out in GEMS™ Version 6.2.3. The model is oriented so that model north is parallel to true north (i.e., no rotation).

The block model comprises blocks measuring 15 m long x 15 m wide x 5 m thick with an origin of:

- x = 603500, y = 5826500, z = 1450
- with 100 columns, 166 rows, and 170 levels.

17.7.1 Interpolation Plan

Block grades were interpolated from the drill hole composites. Ordinary Kriging was applied to blocks in one of the high-grade domains and to the blocks outside of the high-grade domains. Inverse Distance weighted to the second power was applied to blocks inside three of the high-grade domains.

Blocks were interpolated in three passes using search ellipses for sample selection of increasing size. The search ellipses used were: 35 x 35 x 15 m for pass 1, 75 x 75 x 30 m for pass 2, and 125 x 125 x 50 m for pass three. Search ellipses were oriented in the plane of bedding. Two different orientations of search ellipses were used based on the change in dip of bedding near the northern portion of the central high-grade zone. Blocks were interpolated in the first and second passes using a minimum of five composites from at least three holes, not more than 12 of the nearest composites, and not more than two composites were used from a single drill hole. Blocks were interpolated in a third pass using a minimum of three composites from at least 2 holes, not more than 12 of the nearest composites, and not more than two composites were used from a single drill hole.

AGP applied a limited range search restriction to high-grade outliers. The influence of the outliers were limited to neighbouring blocks only, using a maximum search radius of 15 m, under the presumption that true outliers generally have restricted physical continuity. Blocks beyond the restricted radius of a composite above the high-grade threshold do not use that value in the calculation of grade. The threshold grade was applied per domain and details are presented in Table 17-4.

Table 17-4: Restricted Outlier Thresholds

Domain	Outlier Threshold Au (g/t)	Number of Composites Above Threshold	Percentage of Composites Restricted
Upper Contact	7	14	0.7%
Greywacke	10	19	1.6%
Lower	5	14	1.4%
North	4	13	0.6%
Upper Argillite	4	14	0.1%
Greywacke	4	32	0.4%
Lower Argillite	4	33	0.2%

Hard boundaries were applied during the interpolation of grade. Blocks within the four high-grade domains were interpolated using only those composites found within their respective domains. Blocks outside of the high-grade domains were interpolated using only those composites found outside of the high-grade domains.

17.7.2 Mineral Resource Classification

Mineral resources were classified in accordance with definitions provided by the Canadian Institute of Mining, Metallurgy, and Petroleum (CIM) as stipulated in NI 43-101. The mineral resources at Spanish Mountain are classified by AGP as Measured, Indicated, and Inferred.

Any block interpolated in Pass 1 or 2 was classified Indicated. If, however, the block was within the Upper Contact high-grade domain and interpolated in Pass 1, it was classified as Measured. Any block interpolated in Pass 3 was classified as Inferred.

Isolated clusters of blocks that met the criteria for Measured category were reclassified to Indicated. Blocks in the northern portion of the deposit, which passed the criteria of Indicated as outlined above, were reclassified as Inferred due to a low confidence in the controls on mineralization. Blocks outside of the area of focused drilling which passed the criteria for classification as Inferred as outlined above, but only locally, were reclassified as potential resource for the purpose of highlighting targets for future drill investigation.

17.7.3 Block Model Validation

AGP also interpolated grade using the inverse distance (ID) and nearest neighbour (NN) methods, and compared these estimates on swath plots. AGP observed no concerns with respect to the swaths.

The block model was also validated by visually inspecting the block model results on section in order to compare with the drill hole composite and raw sample data. The grades of the blocks by section agreed well with the drill hole data.

17.7.4 Global Mineral Inventory Tabulation

For the purpose of this report, a global mineral inventory is presented in Table 17-5 to Table 17-8. For tabulation of resources, see Section17.7.5.

Table 17-5: Global Inventory – Measured

Cutoff Grade (g/t)	Density (t/m^3)	Tonnes (t)	Average Grade (g/t)	Contained Gold (oz)
0.6	2.78	4,050,100	1.172	152,600
0.5	2.78	4,400,400	1.123	158,900
0.4	2.78	4,656,800	1.086	162,600
0.3	2.78	4,800,700	1.064	164,200
0.2	2.78	4,878,900	1.051	164,900

Table 17-6: Global Inventory – Indicated

Cutoff Grade (g/t)	Density (t/m^3)	Tonnes (t)	Average Grade (g/t)	Contained Gold (oz)
0.6	2.78	23,077,800	1.007	747,200
0.5	2.78	31,403,200	0.885	893,500
0.4	2.78	44,720,100	0.754	1,084,100
0.3	2.78	68,582,900	0.612	1,349,500
0.2	2.78	121,074,900	0.452	1,759,500

Table 17-7: Global Inventory – Measured + Indicated

Cutoff Grade (g/t)	Density (t/m^3)	Tonnes (t)	Average Grade (g/t)	Contained Gold (oz)
0.6	2.78	27,127,900	1.032	899,800
0.5	2.78	35,803,600	0.914	1,052,400
0.4	2.78	49,376,900	0.786	1,246,700
0.3	2.78	73,383,600	0.641	1,513,700
0.2	2.78	125,953,800	0.475	1,924,300

Table 17-8: Global Inventory – Inferred

Cutoff Grade (g/t)	Density (t/m^3)	Tonnes (t)	Average Grade (g/t)	Contained Gold (oz)
0.6	2.78	23,900,400	0.907	697,000
0.5	2.78	35,218,800	0.791	895,700
0.4	2.78	54,330,900	0.669	1,168,600
0.3	2.78	89,978,200	0.540	1,562,100
0.2	2.78	170,395,600	0.400	2,191,300

17.7.5 Mineral Resource Tabulation

Mr. Waldegger exported the completed block model for further economic analysis by AGP's principal mine engineer. The model was exported in an ASCII format with the following items included in that file:

- X, Y, Z coordinates for each block center
- gold grade (g/t)

- rock type
- specific gravity
- classification (1=Measured, 2=Indicated, 3=Inferred).

The mining model was created using the mining software MineSight©. This software was chosen to make use of the integrated Lerch Grossman routine for pit shell development. The model bounds of the geologic model were smaller than what was used for mining by 150 m on both sides in the east-west direction. The model was expanded in MineSight© with default specific gravities (SG) being applied to those blocks. The default SGs applied were:

- Overburden = 2.2 tm^3
- Rock = 2.78 t/m^3

This permitted sufficient width for pit slope designs.

The geologic model grade is based on a whole block basis. This philosophy remains the same for the mining model. An ore percent item was used in the mining model to ensure that blocks at the overburden contact were not improperly accounted. This ensured that the tonnage reported matched rock tonnage correctly.

The mining model was used in the development of the final pit design for this study. The design was completed with proper berm designs, ramps, and access considerations. The use of the Lerch-Grossman shells was only to guide the development of the pit design and not report the resource. The boundaries of the pit are based on the calculated mining cutoff, which considers all mining, processing, and G&A costs. As well, the mill recovery and downstream costs associated with final preparation of the gold are included.

Industry practice is becoming the statement of resources constrained within a pit shell in the case of open pit properties such as Spanish Mountain. While additional mineralized tonnage exists in certain areas, and in particular below the pit, this current exercise did not show potential economic viability with the gold price assumption considered to define the pit shell. Future work at higher gold prices and/or with additional drilling may allow these zones to be included in future resource statement, but for the purpose of this study have not been included. They represent the future potential of Spanish Mountain.

While the pit boundaries are defined with the mining cutoff, within the final pit design the milling cutoff was used for reporting the contained resource. This cutoff is referred to as the milling cutoff as it only considers the cost to process the resource. Material in the design must be moved out of the pit anyways so if the value of the contained gold can pay for milling and all downstream costs, then the tonne of material has met the milling cutoff or exceeded it. The milling cutoff calculated for Spanish Mountain was 0.196 g/t. This has been rounded to 0.2 g/t for reporting purposes.

The resource reported in the pit design for scheduling purposes has been based on the use of diluted grades. The calculation method for determining the dilution has been discussed in Section 19.2.4 of this report. The reader is asked to refer to that for clarification of the dilution calculation. The overall dilution percentage was calculated to be 3.4%

The method employed does not increase the tonnage as the resource loss tonnage is assumed to be the same as the dilution tonnage.

Table 17-9: Spanish Mountain Resource Tabulation by Cutoff with In Situ and Diluted Grade

Resource Category	Units	0.2 g/t	0.3 g/t	0.4 g/t	0.5 g/t	0.6 g/t
Measured	tonnes	4,875,900	4,794,600	4,647,600	4,381,800	4,009,600
In Situ Gold Grade	g/t	1.05	1.06	1.09	1.12	1.18
Diluted Gold Grade	g/t	1.04	1.05	1.07	1.11	1.16
Indicated	tonnes	72,498,800	45,000,600	31,611,200	23,484,600	17,767,200
In Situ Gold Grade	g/t	0.52	0.68	0.82	0.94	1.06
Diluted Gold Grade	g/t	0.50	0.66	0.79	0.91	1.02
Measured + Indicated	tonnes	77,374,700	49,795,200	36,258,800	27,866,400	21,776,800
In Situ Gold Grade	g/t	0.55	0.72	0.85	0.97	1.08
Diluted Gold Grade	g/t	0.53	0.69	0.82	0.94	1.05
Inferred	tonnes	39,531,300	26,133,200	18,366,900	12,564,100	8,534,200
In Situ Gold Grade	g/t	0.48	0.60	0.70	0.81	0.92
Diluted Gold Grade	g/t	0.47	0.58	0.68	0.78	0.90

Table 17-9 outlines the resource tonnes and grade within the 40,000 t/d pit by cutoff grade.

The conversion of mineral inventory to resource is shown in Table 17-10 is for information only.

The mineral resource for Spanish Mountain is shown in Table 17-11 at the cutoff grade of 0.2 g/t.

Table 17-10: Mineral Inventory to Resource Conversion (In Situ Grades Only – 0.2 g/t Cutoff)

Resource Category	Units	Mineral Inventory	Mineral Resource	Percentage of Inventory in Design Pit
Measured	tonnes	4,878,900	4,875,900	100%
In Situ Gold Grade	g/t	1.05	1.05	-
Indicated	tonnes	121,074,900	72,498,800	60%
In Situ Gold Grade	g/t	0.45	0.52	-
Measured + Indicated	tonnes	125,953,800	77,374,700	61%
In Situ Gold Grade	g/t	0.48	0.55	-
Inferred	Tonnes	170,395,600	39,531,300	23%
In Situ Gold Grade	g/t	0.40	0.48	-

Table 17-11: Reported Spanish Mountain Resource Tabulation (0.2 g/t Cutoff)

	Units	Measured	Indicated	Measured + Indicated	Inferred
Resource Tonnage	tonnes	4,875,900	72,498,800	77,374,700	39,531,300
Diluted Gold Grade	g/t	1.04	0.50	0.53	0.47

18 OTHER RELEVANT DATA AND INFORMATION

In November of 2007, Knight Piésold (KP) completed a preliminary TMF alternatives study (Cont. No. VA07-01585) which identified several potential TMF sites located within 15 km of the Spanish Mountain deposit. The suitability of these sites was evaluated based on proximity to the deposit, storage efficiency, and catchment area. Sites located within Cedar Creek watershed were identified as preferred, based on these criteria.

Subsequently, an additional waste and water management alternatives study (Ref. No. VA102-272/5-1 Rev. 0) was completed in August 2010 to optimize and further investigate a number of TMF sites presented in the preliminary study plus an additional site located on the height of land between the Cedar Creek and Spanish Creek drainages to the west of the deposit area. This study was completed to provide a more detailed basis for selection of a preferred TMF location for the Preliminary Economic Assessment (PEA). This study assessed six waste and water management alternatives, which included various components such as the TMF, tailings distribution and reclaim water systems, diversion ditches, seepage collection ponds, haul roads, and the plant site. Preliminary order of magnitude initial capital, sustaining capital, and operating comparative cost estimates were prepared for each alternative to identify a preferred waste and water management option for further study and inclusion in the PEA. The results of this study indicated that Option H1, which included a TMF within the Cedar Creek watershed, was the preferred concept from a cost perspective.

18.1 Waste Rock Management

18.1.1 Waste Rock Production

AGP developed a 40,000 t/d production schedule which defined the amount of ore and waste rock produced on a yearly basis over the mine life, including identification of the type of waste rock produced based on the NP/AP ratio (see Sections 19.3.4 and 22.3).

A simplified waste rock production schedule was developed for the waste and water management study that assumes that 14 Mt of the PAG waste rock produced will be disposed in the TMF. The remaining PAG material will be comingled with the NAG in the various waste dumps or placed in the backfill of Phase 2 of the mining plan.

18.1.2 *Waste Disposal Strategy*

Three waste types are generated as part of the mining process:

AG – Acid Generating

PAG – Potentially Acid Generating

NAG – Non-Acid Generating

Less than one percent of the waste material has been estimated to be AG in character. The waste classification model is rudimentary for this study though and requires further study. The indication provided by the existing test work is that the predominant waste type is PAG, which represents 72% of all waste material. The remainder is NAG.

Five separate dump locations will be developed for disposal of waste material. These are:

1. TMF
2. West Dump
3. East Dump
4. Plant Dump
5. Phase 2 Backfill.

All AG material will be stored in the TMF. Suitable NAG waste rock will be hauled from the open pit to the TMF for use in dam construction. The distance from the open pit to the centre of the TMF is approximately 4 km.

The PAG waste rock will be deposited within the TMF in such a manner that it is progressively encapsulated by rougher scavenger tailings and saturated by the supernatant pond.

The West and East dumps are on the periphery of the open pit. They will contain both PAG and NAG material comingled.

The Plant dump is located in the Cedar Creek valley adjacent to the plant facility. It will contain a mixture of PAG and NAG material from the open pit.

The Phase 2 Backfill will take all waste material from the open pit from Year 7 onwards when Phase 2 mining is completed. This will include PAG and NAG. The total tonnage of waste stored in the backfill will be 20% of the waste or 45.6 Mt.

Concurrent reclamation is planned for the waste dumps with resloping and revegetation.

18.2 Tailings Management Facility

18.2.1 General

The preferred waste and water management alternative identified for further study and inclusion in the PEA was Option H1, which included a TMF located within Cedar Creek to the south of the deposit. This was decided upon after multiple locations were considered. This option was further studied and optimized for inclusion in the PEA; the resulting arrangement is called Option H2. Figure 18-1 shows the general arrangement of Option H2.

18.2.2 Design Basis and Operating Criteria

The principal objective of the TMF is to provide secure containment of all tailings solids and PAG waste rock.

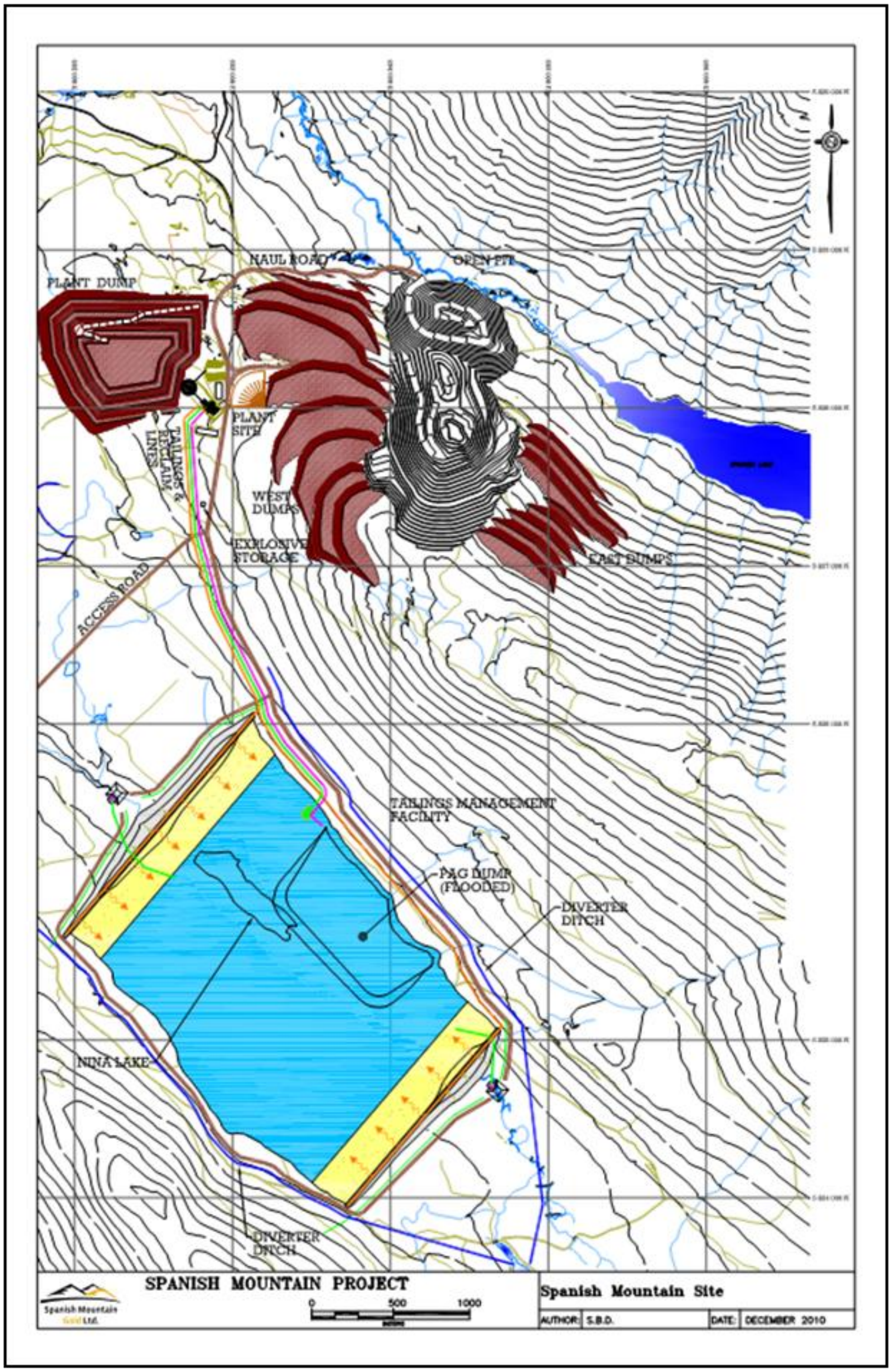
The mill throughput is 40,000 t/d, with a total of 116 Mt of ore milled over the 10 year life-of-mine. The concentrate is assumed to be 1% of the total ore milled, with the remaining 99% discharged to the TMF.

The metallurgical process involves a gravity circuit followed by a rougher flotation circuit to produce Rougher Scavenger Tailings (RST). Approximately 10% of the RST will be reground and subjected to a cleaner flotation circuit to produce cleaner scavenger tailings (CST), which are assumed to be acid generating if allowed to oxidize. The tailings streams will be transported from the plant site to the TMF in separate pipelines at an average solids content of 30% by weight. Each tailings stream will be deposited independently; the RST will be discharged along the TMF embankments to create tailings beaches, and the CST will be discharged to allow for progressive encapsulation by the RST and saturation by the supernatant pond.

The TMF capacity at all stages of the mine life includes the supernatant pond volume and allowances for wave run-up (0.5 m), post-seismic settlement (0.5 m), sloping beaches (2.5 m), and containment of the inflow design flood (4 Mm^3).

A simplified waste rock production schedule was developed for the waste and water management study; it was assumed that 14 Mt of PAG waste-rock would be produced over the mine life, requiring disposal within the TMF.

Figure 18-1: Spanish Mountain Site Layout



18.2.3 Tailings Management Facility Embankments

The TMF includes two water-retaining, zoned earthfill/rockfill structures with a low permeability core and an appropriate downstream filter relationship.

The starter TMF will be constructed during the pre-production phase and is sized to store the estimated volume of tailings and PAG waste rock produced during the first two years of operation, plus the supernatant pond volume and allowances for wave run-up, post-seismic settlement, sloping beaches, and containment of the inflow design flood. The TMF embankments will be constructed in annual stages with each stage providing the required capacity for the period until the next stage is completed, with a final storage capacity of approximately 126 Mt of tailings, 14 Mt of PAG 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 centreline construction methods while maintaining a 2.25:1 downstream slope.

18.2.4 Tailings Distribution and Reclaim Water Systems

The RST will be discharged into the TMF from a series of large-diameter valved offtakes 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 PAG waste rock and CST.

The CST will be discharged separately to allow for progressive encapsulation by the RST and saturation by the supernatant pond.

Process water will be reclaimed from the TMF supernatant pond using barge-mounted pumps and a dedicated reclaim water pipeline.

18.2.5 Water Management

The TMF supernatant pond serves as a primary component in site water management, providing a buffering for process water, direct precipitation, and runoff.

Surface diversion ditches have been included to capture and divert non-contact water around the TMF for release to the environment. The water will be diverted to Boswell Lake, where it will flow 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.

18.3 Haul Roads

Allowance for the mine haulroads is 33 m wide. This includes 23 m for a running surface and a 1 m ditch on either side if required. Where berms are required in accordance with the BC Mining Regulations, the berms will have a base of 7.7 m and a height of 2.6 m. That provides a berm height of 3/4 the largest tire.

Road grades will be 10% within the pit. A maximum of 8% downhill out of the pit to the plant is considered.

All ditches from the haulroads will be directed to the tailings facility with appropriate culverts, or to small settling ponds where sediment will be allowed to collect from surface runoff.

At mine closure, these roads will be cross-ditched for drainage and the road surface scarified and revegetated.

18.4 Plant Site

The plant site drainage will be collected in a settling pond with disposal to the tailings facility. Wash bay drainage will be directed to an adjacent settling pond and pumped to the TMF.

19 MINING AND GEOTECHNICAL

19.1 Geotechnical and Engineering Geology Assessments

BGC's scope of work included a site visit, examination of drill core, and compilation of geotechnical and structural geologic data made available by Spanish Mountain. This information has been used to provide scoping level geotechnical assessments for a potential open pit at the Spanish Mountain property.

The preliminary engineering geology interpretations and geotechnical studies presented in this report have been developed based on the following information provided by Spanish Mountain Gold:

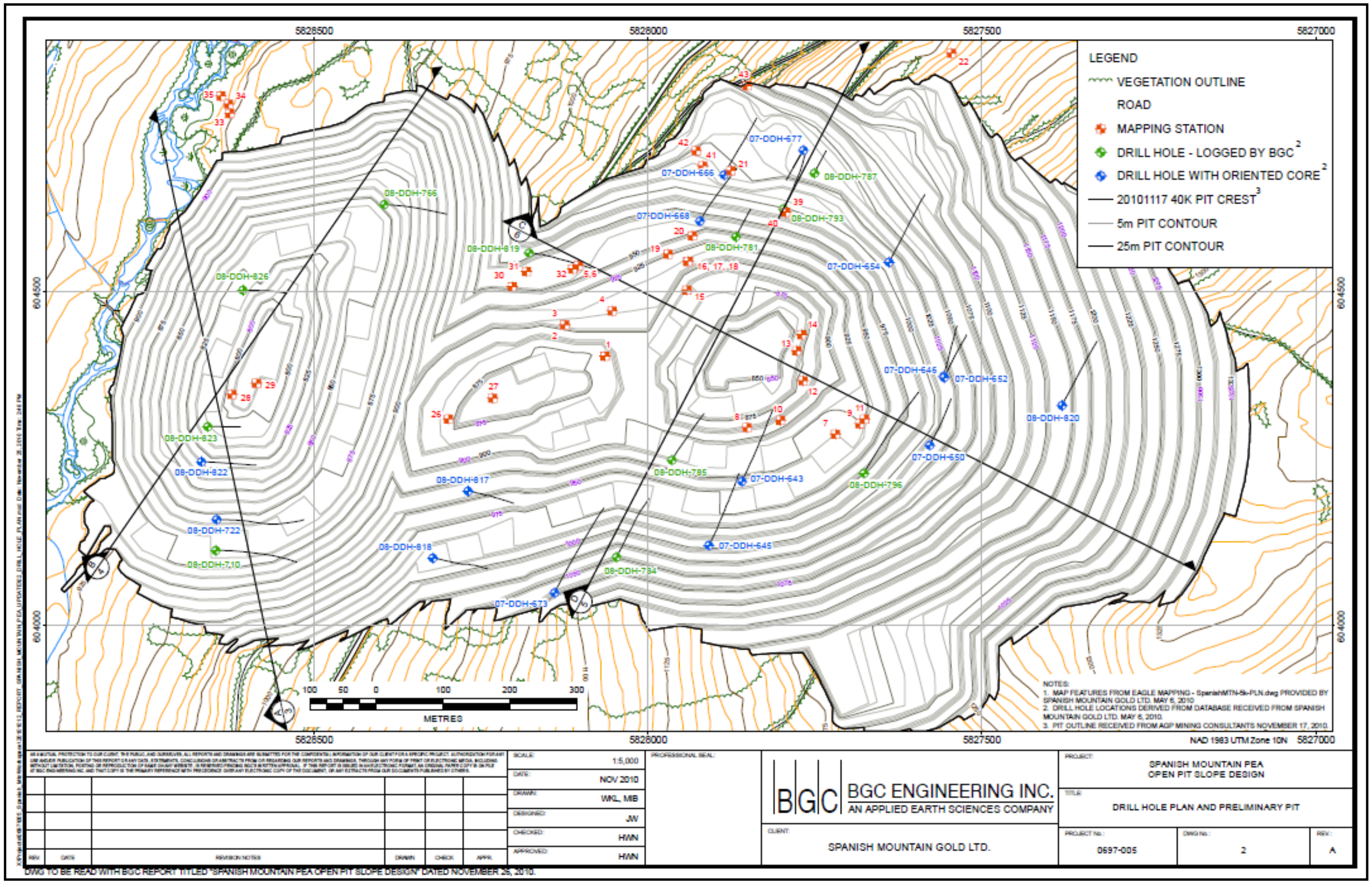
- Site geology maps and sections included in a Technical Report on Resource Estimation on the Spanish Mountain Gold Deposit; NI 43-101 Technical Report – Skygold Ventures Ltd., 2009).
- Preliminary geotechnical data collected by Spanish Mountain staff during logging, including core recovery, RQD, hardness, fracture count, lithology and alteration data.
- A report entitled "Structural Interpretations" by Georgina Price, M.Sc. P. Geo., which was based on 2006 diamond drilling and surface mapping by Skygold Ventures Ltd., and trench mapping by Cyprus Canada Inc., Mt Calvery Resources Ltd., and Wildrose Resources Ltd., dated January 2008.
- Core photographs taken by Spanish Mountain.

The recommended pit slope angles were developed based on the following tasks conducted by BGC:

- rock mass characterization of select core intervals
- review local and regional structural geology
- compilation of mapping and oriented core data
- review of rock and alteration types with Spanish Mountain geologists.

In total, 830 m of core from the Spanish Mountain deposit area were examined. Details regarding the drill hole identities and intervals of holes logged are included in Appendix C. The locations of the drill holes logged, and the surface outcrops mapped by Spanish Mountain are shown in Figure 19-1.

Figure 19-1: Drill Hole Plan and Preliminary Pit



Spanish Mountain Gold has completed five hundred and eighty six (586) exploration core holes in the vicinity of the deposit, with a total of 68,195 m of core drilled to date. Rock Quality Designation (RQD) was logged by Spanish Mountain for holes drilled since 2007, with fracture count and hardness added to the core logging information collected in 2009. BGC logged additional geotechnical parameters on selected intervals of split (sawn) core during the site visit. The locations of the holes which BGC partially logged are shown on Figure 19-1. A geotechnical database has been developed which includes data from work completed by Spanish Mountain Gold, KP, regional geology mapping, and more recent work undertaken by BGC. The information in the database was used to estimate geomechanical design parameters for the proposed open pit. The rock mass was divided into geotechnical units for design purposes, with the geotechnical units grouped according to lithology (i.e., rock type).

A relatively limited amount of structural data has also been collected from regional geologic mapping, outcrop mapping and oriented core drilling conducted by Spanish Mountain. The locations of structural measurements collected by Spanish Mountain, from both surface exposures and from oriented core measurements are presented on Figure 19-1.

Data provided to BGC during the site visit has been plotted on equal area, lower hemisphere stereonets. Structural discontinuity data from the Main and North zones have been separated. The quantity of structural data is limited, particularly for fractures and faults; however, sufficient data is available for preliminary structural fabric interpretations of the rock which will be encountered in the proposed open pit. Bedding measurements are relatively abundant and have been assumed to be more reliable and predictable due to regional mapping efforts undertaken in the area.

19.2 Slope Design Methodology

There are two main controls on achievable open pit slope design angles. The first consideration is the potential for structural instabilities, whereby discontinuities in the rock mass (joints, bedding planes, faults, and other) intersect the excavation such that it becomes “kinematically possible” for failure to occur, i.e., the geologic discontinuities daylight out of the slope. Achievable slope angles are therefore limited by the orientation and the shear strength of the discontinuities. Structurally controlled slope failures can occur at any scale, i.e., at the bench, inter-ramp, and the overall slope scales.

The second consideration is the strength of the rock mass. This is dictated by the amount of fracturing within the rock mass, the characteristics of the discontinuities, and the intact rock strength. Rock mass stability generally includes large-scale, deep-seated failures and slope-scale failures through weak geological units.

19.2.1 *Structural Control on Pit Wall Stability*

The structural geologic model of the Spanish Mountain deposit and surrounding area is still being developed, and thus relatively limited information was available at the time the PEA was undertaken. As a result, only a cursory evaluation of the impacts of geologic structure on pit wall stability could be undertaken. Structural data collected from surface mapping and oriented core drilling were plotted on lower hemisphere equal area stereonets. Kinematic stability assessments were carried out using the main structural discontinuity sets identified on the stereonets.

19.2.2 *Rock Mass Strength Assessments*

Rock mass strength parameters have been assigned to the three primary geological units (siltstone, argillite, and greywacke). For the generic stability analyses, the rocks comprising the pit walls were assumed to be homogeneous with no structural controls, i.e. the site specific geology was not incorporated into the cross-sections, to simplify the stability analyses. Generic stability analyses were carried out to evaluate various pit wall geometries, with slope heights ranging from 100 m to 500 m, and overall pit wall angles ranging from 30° to 60°. The results of the generic stability analyses can be used to provide broad guidance to mine planners on achievable overall slope angles within the primary rock types.

Hydrogeologic conditions for the Spanish Mountain deposit are not well defined. However, at this preliminary economic assessment stage it has been assumed that the slopes have been completely dewatered. Therefore, the generic, rock mass stability analyses have been conducted assuming dry conditions, i.e., with a pore pressure coefficient (Ru) of zero. Relatively low inter-ramp heights of 200 m and 100 m have been assumed for the Main and North Zones, respectively, to facilitate aggressive depressurization of the pit walls by allowing frequent dewatering well installation as the pit is deepened.

Factors of safety (FOS) were calculated for various slope heights and angles. Slope height/slope angle combinations resulting in a FOS between 1.2 and 1.3 have been developed to determine acceptable slope angles for given slope heights under both dry and partially saturated conditions.

19.2.3 *Recommended Pit Wall Design Angles*

Kinematic stability analyses and rock mass failure analyses have been compared to determine which will limit the inter-ramp slope angles. In some sectors of the proposed pits the pit wall geometry is also limited by the geometry of the benches due to regulatory requirements. The maximum inter-ramp angles for each of these evaluations are summarized in Table 19-1. The design angle presented is the lowest (i.e., limiting) of the three values.

Note that maximum inter-ramp slope heights of 100 m and 200 m have been assumed for the North Zone and Main Zone, respectively. This is the maximum allowable height between a ramp or a "geotechnical berm." A geotechnical berm should be at least a half ramp width to accommodate dewatering wells and/or geotechnical instrumentation to track the progress of slope depressurization and monitor slope performance. Also, note that the maximum angles indicated for potential rock mass failure are based on the estimated overall slope height for that design sector, as shown in Figure 19-1. Should the overall height vary significantly from those indicated in Table 19-1, the maximum allowable angle to avoid rock mass failure may change. In particular, if the overall height increases the allowable angle will decrease.

19.3 Open Pit Mining

19.3.1 Introduction

The objective of the Preliminary Economic Assessment is to evaluate the potential economics of an open pit operation at Spanish Mountain. This considers the latest drilling and geologic interpretation with a gold price representative of the last three-year average for pit design work. The study also provides guidance on further work for Spanish Mountain Gold.

Unless otherwise noted, all prices are in 3Q 2010 Canadian dollars.

19.3.2 Geologic Model Importation

AGP updated the geologic model with the latest drilling and interpretation as noted in Section 17 of this report. The geologic model was created using the Gemcom© software package. For mine planning work, MineSight© was used, which required a transfer of the model information.

This was accomplished using an ASCII file export of the geologic model. This file contained:

- X, Y, Z coordinates for each block centre
- gold grade
- rock type
- specific gravity
- ore classification (1= Measured, 2= Indicated, 3= Inferred).

Table 19-1: Recommended Bench and Inter-ramp Configurations

Domain	Design Sector	Slope Azimuth Start (°)	Slope Azimuth End (°)	Maximum Inter-ramp Height [1] Ih (m)	Bench Height[2] Bh (m)	Bench Face Angle[3] Ba (°)	Bench Width Bw (m)	Maximum Inter-ramp Angle: Bench Geometry[4] (G) Ia (°)	Maximum Inter-ramp Angle: Kinematic[5] (K) Ia (°)	Maximum Inter-ramp Angle: Rock Mass[6] (R) Ia (°)	Design Value Ia (°)	Approximate Overall Slope Height[7] Oh (m)
Main Zone	MZ-010	315	065	200	20	65	9.5	47	-	47	47	125
	MZ-108	065	150	200	20	65	9.5	47	-	47	47	225
	MZ-180	150	210	200	20	65	17.5	37	39	47	37	425
	MZ-263	210	315	200	20	65	9.5	47	-	47	47	300
North Zone	NZ-025	345	065	100	10	65	9.0	36	43	53	36	125
	NZ 128	065	190	100	10	65	9.0	36	-	53	36	225
	NZ-238	190	285	100	10	65	9.0	36	39	53	36	225
	NZ-315	285	345	100	10	65	12.0	31	32	53	31	125

Notes: **1**. Maximum inter-ramp height assumed based on typical pit dewatering and geotechnical instrumentation requirements. Lower inter-ramp heights of 100 m are required in North Zone due to the proximity to Spanish Creek.
2. Bench height provided by AGP Mining.
3. Bench face angle assumed based on average angle from BGC database of bench geometries.
4. Geometric control based on bench height = 20 m in Main Zone and 10 m in North Zone. Bench face angle = 65° and bench width as shown.
5. Inter-ramp slope angles limited by kinematic controls are based on the maximum angles that can be obtained without undercutting bedding, where bedding dips greater than 30°. Bedding design sets are indicated in stereonets for the Main Zone and the North Zone in Appendix C.
6. Maximum allowable inter-ramp angle due to rock mass quality based on assumed maximum inter-ramp height in typical rock type for that domain. Inter-ramp and maximum slope angles are based on assumed fully depressurized slopes (Ru=0)
7. Height estimated from pit plans provided by AGP Mining Consultants, 17 November 2010. Design curves should be used to determine maximum overall slope based on the overall slope heights, ensuring that the maximum inter-ramp heights are not exceeded.

The model bounds of the geologic model were smaller than what was used for mining by 150 m on both to the east and west direction. The model was expanded in MineSight© with default specific gravities (SG) being applied to those blocks.

The default SGs applied were:

- overburden = 2.2 t/m^3
- rock = 2.78 t/m^3

This permitted sufficient width for pit slope designs.

The geologic model grade is based on a whole block basis. This philosophy remains the same for the mining model. An ore percent item was used in the mining model to ensure that blocks at the overburden contact were not improperly accounted. This ensured that the tonnage reported matched rock tonnage correctly.

Subsequent to the importation of the grade model, an ARD model was developed and loaded into the appropriate mining model item. This will be discussed later in the mining section as it was used in the mine schedule.

19.3.3 Production Rate Trade-off Study

The mining model created provided the basis upon which to examine the Spanish Mountain deposit for development. Due to the shallow deposit configuration and lower grade nature, open pit mining was considered the most reasonable approach. The production rate, which maximized the NPV of the deposit, needed to be determined and a production rate trade-off study was initiated. The intent of this study was not to fully define the costs associated with mining and processing, but rather to examine options at an order of magnitude level to determine what was reasonable with the known geology.

Design Criteria

Gold price volatility existed throughout the period of the study, making precise metal prices for long-term use a difficult choice. An approximate three-year rolling average gold price was decided on for the study. Spanish Mountain Gold and AGP agreed for the purposes of the trade-off study and any pit designs to use the following parameters:

- gold price = US$950/oz
- exchange rate = C$1.10 : US$1

These prices formed what was termed the engineering base case prices and would be used for all design purposes. AGP and Spanish Mountain Gold considered them to be conservative relative to the three year average price which was approaching US$1,000/oz and spot prices in excess of US$1,200/oz.

Metal recoveries were available from testwork completed prior to initiation of the PEA study. Testwork also continued during the PEA study, which further confirmed the recovery and operating cost estimates. For the production rate trade-off study and the final design, a gold recovery of 90% was used.

Refining charges for the gold were estimated at US$8/oz with 98.5% payable to Spanish Mountain Gold.

Labour costs were developed considering current collective agreements at Imperial Metals neighbouring Mt. Polley Mine and the Gibraltar Mine owned by Taseko. These were applied to the various mining, process, and general and administrative (G&A) functions.

Fuel was estimated at a cost of C$0.73/L. Electrical power was estimated to be C$0.04/kWh.

Mine equipment operating costs are discussed later, but were developed from quotations of major vendors for equipment, explosives, and tires.

Trade-off Study Cost Estimates

The production rate trade-off study focused on production rates between 20,000 and 40,000 t/d of mill feed with a 5,000 t/d increment and 50,000 t/d. AGP determined that this was a valid range considering the size of the deposit in the updated model.

Mining costs were estimated using haulage profiles with waste material being hauled to the tailings option, H2, which is in the next valley to the south of the deposit. To determine the costs, a truck size of 181 tonnes was assumed with appropriate-sized hydraulic shovel. For waste and plant feed movement, a strip ratio of 1.7:1 was considered.

At the time of the trade-off study, ARD testwork results were still pending. An assumption was made by AGP, Spanish Mountain Gold, and Knight Piésold to assume that 75% of the material was acid generating. This meant that 75% of the waste material would have to be stored within the tailings facility. The remaining 25% could be placed in close proximity to the pit being mined.

The processing cost included the cost of moving tailings to the H2 option.

G&A costs were estimated based on the tonnage mined and costs reported by both Gibraltar and Mt. Polley mines. AGP considered these costs to be indicative of what would be expected in the final operation due to the proximity of these mines.

Table 19-2 summarizes the cost inputs used in the production rate trade-off study.

Table 19-2: Trade-off Study Input Parameters

	Mill Production Rate (t/d)					
	20,000	25,000	30,000	35,000	40,000	50,000
Mining Cost – $/t mined	2.15	2.08	2.00	1.94	1.89	1.84
Processing – $/t (mill feed)	6.08	5.83	5.66	5.50	5.43	5.36
G&A – $/t (mill feed)	0.70	0.59	0.56	0.54	0.52	0.42

Pit Shell Development and Mine Schedule

Geotechnical parameters, discussed in the preceding section, were included in the mining model. Once factored, they could be applied to the development of pit shell slopes. A series of eight different domains were coded, four each for the North Zone and the Main Zone. Their locations have been indicated in Figure 19-2. The overall angles used are shown in Table 19-3.

Figure 19-2: Mining Geotechnical Domains

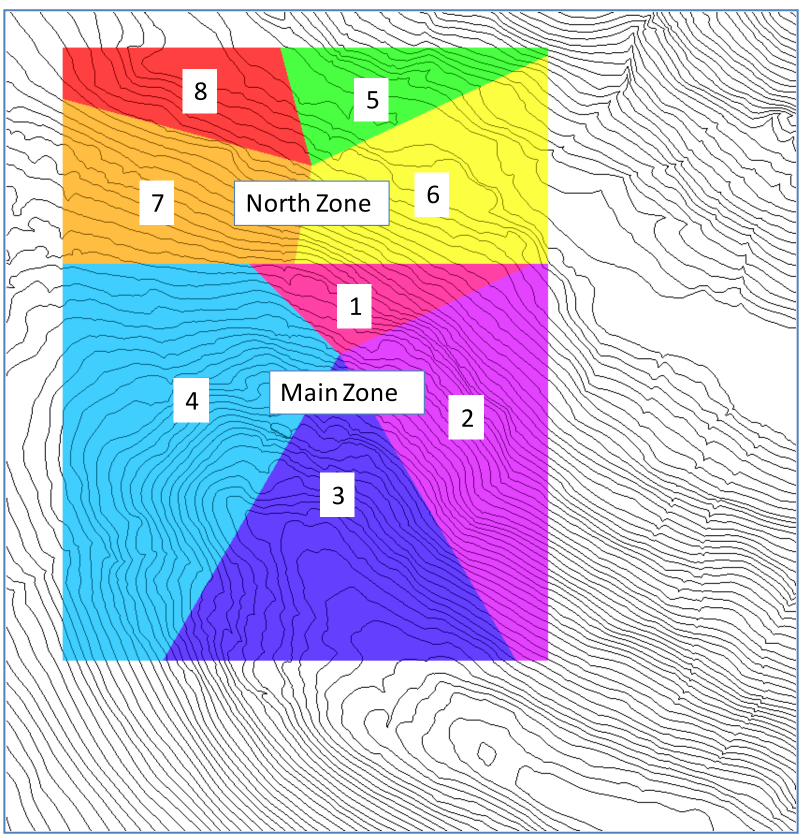


Table 19-3: Mining Geotechnical Domain Overall Angles

	Main Zone				North Zone			
Domain	1	2	3	4	5	6	7	8
Overall Angle (°)	43	43	37	43	36	36	36	31

The overall angles were determined based on a quick pit using the initial geotechnical parameters. The pit depth and length was considered to determine placement and number of ramps that would be expected to be completed. Ramp widths of 33 m were applied. Areas 3 and 8 in Figure 19-2 were excluded from having ramps due to the already low angle expected. The others were flattened to reflect expected final overall wall slopes. These are also shown in Table 19-3.

These parameters were used with the Lerch-Grossman routine, bundled as part of the MineSight© software. With this, the ultimate pit shells for the various production cases were developed.

To determine intermediate shells for phasing of the pits, the base gold price was reduced and the pit shells run with the same operating costs. The gold price was varied from US$950/oz to a low of US$450/oz. Using a minimum mining width of 80 m, phasing for each production rate case was established. This resulted in three phases for each case being mined. No separation between the Main Zone and the North Zone was considered in the trade-off schedules. Tonnes and grades within each phase were tabulated and scheduled to achieve the required production rate being examined. The mine schedules for each of the options were then placed in a cashflow model to examine each option relative to one another.

Capital costs for mining, processing, and infrastructure (including tailings) were developed at an order-of-magnitude level. These were applied to the applicable production rate. In the case of the mining equipment, estimates of annual production capability were established to determine the quantity of equipment required.

These were:

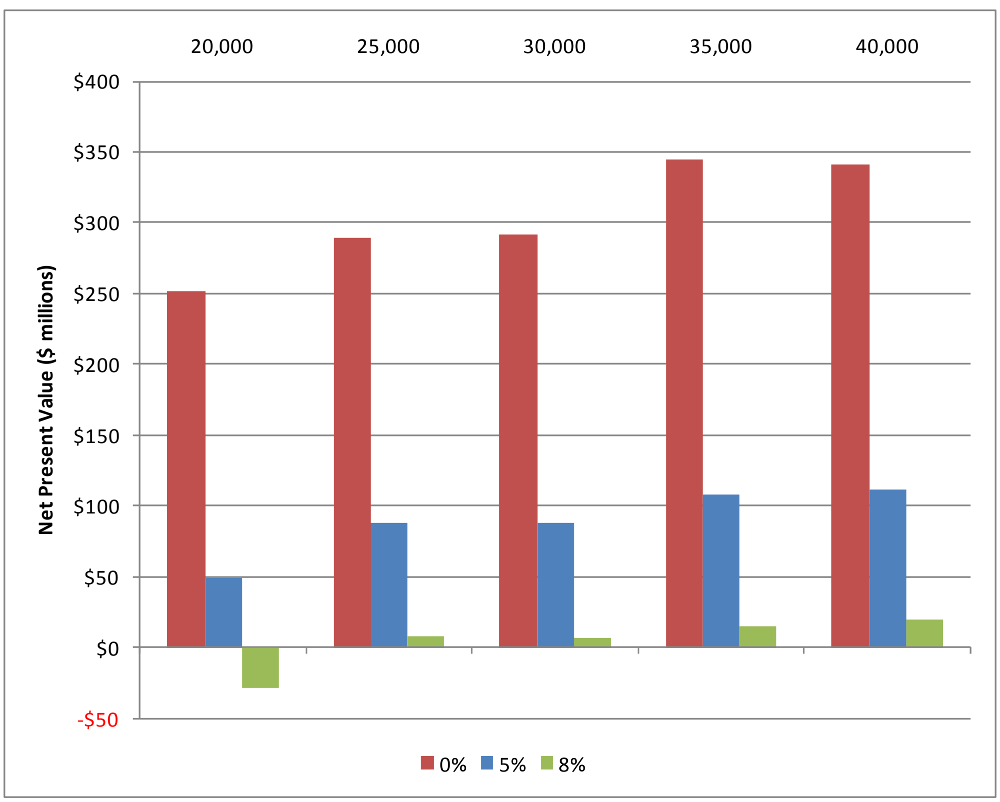
- Drills – 15 Mt/drill/year
- Trucks – 2 Mt/truck/year
- Shovel – 14.4 Mt/shovel/year
- Loader – 10 Mt/loader/year.

Capital unit costs came from vendor budgetary quotations.

Realizing that very approximate numbers are applied in the trade-off study, it is quite often normal to see NPV values being negative. What is important in this stage is to determine the relative differences between the various options, and to highlight the one that best requires detailing in the PEA study. The Spanish Mountain trade-off study was the same, with the initial results all negative. This was attributed to early estimates for all disciplines in operating costs. In the case of mining this was impacted by the percentage of material hauled a greater distance to the tailings facility.

The results of the initial pass using the Engineering Base Case gold price of US$950/oz indicated that the 25,000 t/d option was the least negative. A sensitivity to gold price was applied to determine if one particular option performed better at higher prices. The prices were varied from US$950 to US$1300/oz. As the gold price increased, the higher production cases were the better choices, with less than $20 million separating the highest NPV and the lowest. The spot metal price of US$1,298/oz at the time of the study was also applied, and the results are shown in Figure 19-3.

Figure 19-3: Trade-off Study Results – US$1,298/oz



These results showed that, with a discount rate of 5%, the 40,000 t/d option was favoured. Spanish Mountain Gold in discussion with analysts, believed that the outlook for higher sustained gold prices was good and, coupled with potential for resource expansion, opted to proceed with the 40,000 t/d rate for detail design.

Detail on the production rate trade-off study has been included in Appendix D.

19.3.4 Pit Design and Phasing – 40,000 t/d

With the direction to advance the 40,000 t/d case determined, each discipline then proceeded to detail their operating and capital cost estimates. The trade-off study highlighted various items in the mining area that might assist in lowering the mine operating cost and capital requirements.

These included:

- waste material classification
- waste dump location
- backfill opportunities.

In the trade-off study, an assumption was made that 75% of the material was ARD and would require deposition in the tailings pond. This impacted the overall project costs by increasing the mine operating cost due to the long downhill haul, the mine capital cost due to the longer haul, and the size of the TMF due to a larger volume of material storage. The testwork on the waste samples indicated that the initial assumption was extremely conservative. Of the 78 samples examined, only four indicated they were net acid generating and three of those were above 0.2 g/t Au, or plant feed material. Potentially acid-generating (PAG) samples represented 35% of the total samples, with an average ratio of NP:AP of 1.97, right in the middle of the potentially acid generating spectrum. The remainder of the samples were net acid neutralizing, with an average ratio of 44.7:1, or highly buffering. Using this information, a simple ARD model was estimated by AGP.

With this information, AGP, Spanish Mountain Gold, and KP decided to send all acid-generating material which was not already sent as plant feed to the tailings facility. It was further decided that a portion of the potentially acid-generating material would also be required to be sent to the tailings facility. It was assumed that 1.4 Mt of PAG material per year would be sent, or approximately 9% of the total PAG material over the life of the mine based on the trade-off study tonnages. This separation is something that will have to be confirmed with additional testwork, but was considered reasonable for the purposes of this study. The decision allowed the design capacity of the tailings management facility to be reduced in addition to a reduction in the haulage requirements from the mine.

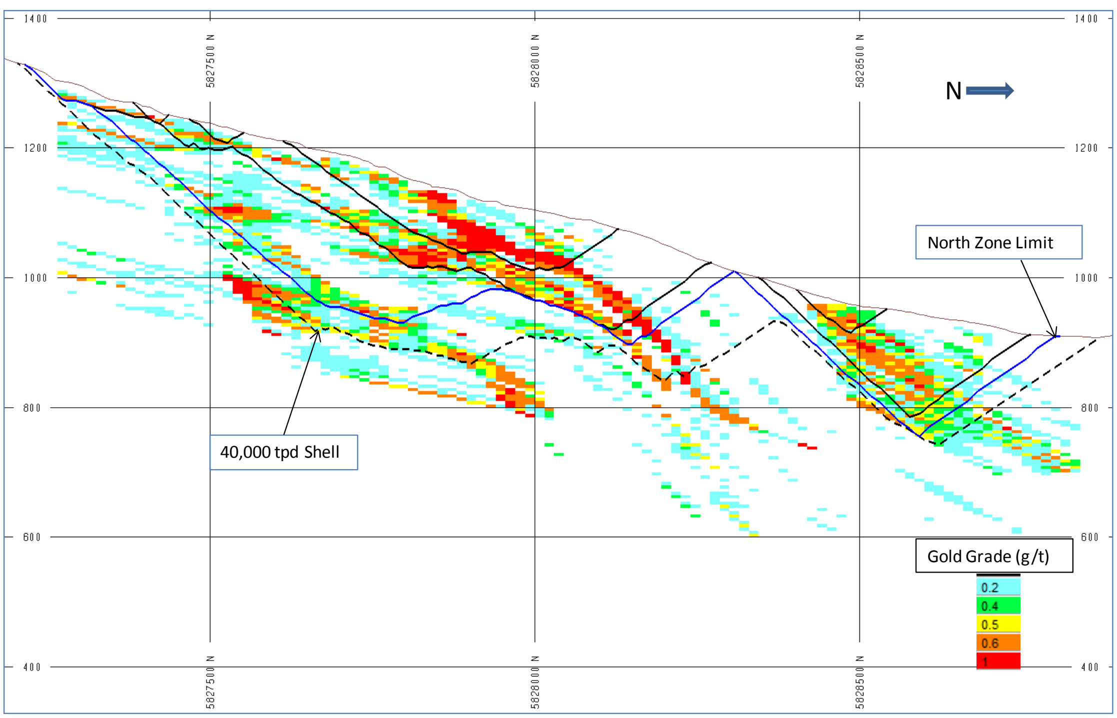
The remaining PAG material would be comingled with the NAG material in normal waste dumps. In addition, NAG material would be used in the construction of the TMF in the initial years.

With the determination of the PAG material, there was more flexibility in determining the waste dump locations. To reduce the operating cost of the mine, waste dumps on the periphery of the pit would be designed.

The third consideration was also in the scheduling of the mine, which affects the pit design and phasing. During the trade-off study, it was noted that the North Zone formed its own pit with the internal wall left behind. This offered an opportunity to backfill if the sequencing would allow. The decision was made to backfill if it made economic sense.

The mine schedules used in the production rate trade-off study mined shells were determined from the Lerch-Grossman routine in MineSight©. These shells were used for the pit design. Detailed examination of the shells showed that the 40,000 t/d ultimate shell would affect on Spanish Creek, with the potential to increase permitting time required for the property. A smaller North Zone pit was considered to avoid interference with Spanish Creek. The 25,000 t/d pit shell for the North Zone was used to define the limits of what that phase of the pit would be. The limit of that shell is shown in Figure 19-4.

Figure 19-4: Pit Shells Used In Pit Design (North South Section at Easting 604340)



The 40,000 t/d LG shell is shown in Figure 19-4 as the black dashed line, clearly indicating in the North Zone the potential wall intersection of Spanish Creek. The thick blue line (next line upwards in the figure) represents the 25,000 t/d final pit outline. In the North Zone the southern slope is similar, but the northern slope does not advance to the north intersecting the creek. For this reason, the 25,000 t/d outline was chosen for the north slope of the North Zone.

In examining the deposit, the decision was made to mine in four phases. This was accomplished by separating out the North Zone pit as a separate entity. The upper two shells represented internal phases and the 40,000 t/d pit shell was the final design outline for the Main Zone.

Due to geotechnical concerns on the south slope of the Main Zone, a conscious decision was made to avoid ramp development on that slope. The design criteria had also indicated that for every 200 m in vertical height for the Main Zone, an extra width or geotechnical berm (a minimum of 17 m) would need to be applied. In the North Zone, the spacing of the geotechnical berms is 100 m vertically.

The deposit bedding parallels the topography in the Main Zone in a dip-slope configuration. This was used in the mine design development as it allows access for waste and ore at various contour elevations. Using this feature of the deposit minimizes the use of ramps on wall slopes in the same manner as used in the coal mines in south-eastern British Columbia. This had the added benefit of reducing waste volumes in the final pit configuration.

Using the geotechnical criteria, pit outline, and discussed design philosophy, four phases were designed. The tonnes and grades have been tabulated in Table 19-4.

Table 19-4: Design Pit Phases – Tonnes and Grades

Phase	Plant Feed (t)	Diluted Au (g/t)	Waste AG (t)	Waste PAG (t)	Waste NAG (g/t)	Total Waste (t)	Strip Ratio
Phase 1	16,989,200	0.72	-	9,491,400	6,287,600	15,779,000	0.9
Phase 2	25,388,000	0.46	12,500	48,148,900	2,498,100	50,659,500	2.0
Phase 3	24,315,800	0.57	3,100	21,414,700	16,441,800	37,859,600	1.6
Phase 4	50,213,015	0.43	-	86,265,500	39,562,500	125,828,000	2.5

Phase 2 in Table 19-4 represents the North Zone pit. Waste has been broken into the three waste categories of:

- AG – Net Acid Generating
- PAG – Potentially Acid Generating
- NAG – Non-Acid Generating or Net Buffering.

Diluted gold grades were quoted in the table. These were determined as contact dilution. The method is to query each block and determine if it meets the cutoff criteria. For the purposes of this calculation, the milling cutoff was used as the cutoff. The milling cutoff is when the revenue generated by the contained gold equals the cost of processing, G&A and all downstream costs. The mining cutoff, which is used to define the pit shape, includes the mining cost as part of the costs that must be covered by the gold revenue.

The Lerch-Grossman routine used by MineSight© can calculate the milling cutoff and deposit the value as an item in the mining model. Both the mining cutoff and the milling cutoff values were stored. When a block is considered to be at or above the cutoff, the surrounding blocks are queried also. The number of blocks below cutoff are stored as an item in the model as the number of diluting sides. The average grade of the below cutoff blocks is also stored as the diluting grade.

The final diluted grade of the block is then based on:

- initial block grade
- number of diluting sides
- average grade of diluting material.

The initial block grades come from the geologic model. The number of diluting sides comes from the query mentioned previously. The average grade of the diluting material also comes from that same initial query.

The number of diluting sides is important as it is used to estimate the percentage of dilution that block would be exposed to. For this project, an estimate of 1 m of dilution per side was considered reasonable with the equipment considered and the block size of 1 5 m wide x 5 m high x 15 m long. This equated to the dilution percentages shown in Table 19-5.

Table 19-5: Dilution Percentages

Diluting Block Sides	Dilution Percentage
0	0.0
1	6.3
2	11.8
3	16.7
4	21.1

When there are no diluting sides on a block, it indicates a condition where the block is surrounded by above-cutoff material and as such would not be subject to waste dilution.

Certainly, grade can move between mill feed blocks, but for the purpose of this study that was neither considered nor estimated.

The dilution percentage increases with the number of diluting sides a block has around it, as would be expected. An isolated block would be subject to a high dilution percentage of 21.1%. This is more representative of actual mining practice than a blanket average dilution percentage. The higher percentage may cause that block to no longer be mill feed grade material.

The diluted grade was then calculated in the following manner:

$$DAu = (100\text{-Dilution }\%) \times Au + (\text{Dilution }\% \times AuWst)$$

Where:

DAu	=	diluted gold grade
Au	=	undiluted gold grade
Dilution %	=	dilution percentage
AuWst	=	gold grade of the diluting material

Each block was examined and the grade in that block was calculated and stored as the diluted grade. The additional waste associated with the dilution was considered to equal the amount of mill feed loss, making a conservative in the estimation of mill feed tonnage. This is strictly diluting the grade, not the tonnes of feed.

The overall average dilution percentage worked out to be 3.4%.

Phase 1

Phase 1 starts half way up the mountain, targeting the first area of high-grade material. Access for this phase like all the phases comes from the western edge, closest to the plant location. From the 1035 level, the pit has a ramp developed down into the higher-grade portion of the phase. Figure 19-5 illustrates the final shape of Phase 1.

Phase 2

Phase 2 is the North Zone pit. This phase was initiated up the slope of the deposit. However, the majority of the deposit is mined from just above the level of Spanish Creek. The access for the pit phase below the creek level remains on the west side. Because Phase 2 intersects a portion of Phase 1, an extra width berm (haulroad width) is left on the 1035 level to continue to provide access into Phase 1 once the Phase 2 mining has progressed below that level. Phase 2 is shown with Phase 1 in Figure 19-6.

Figure 19-5: Phase 1 Design

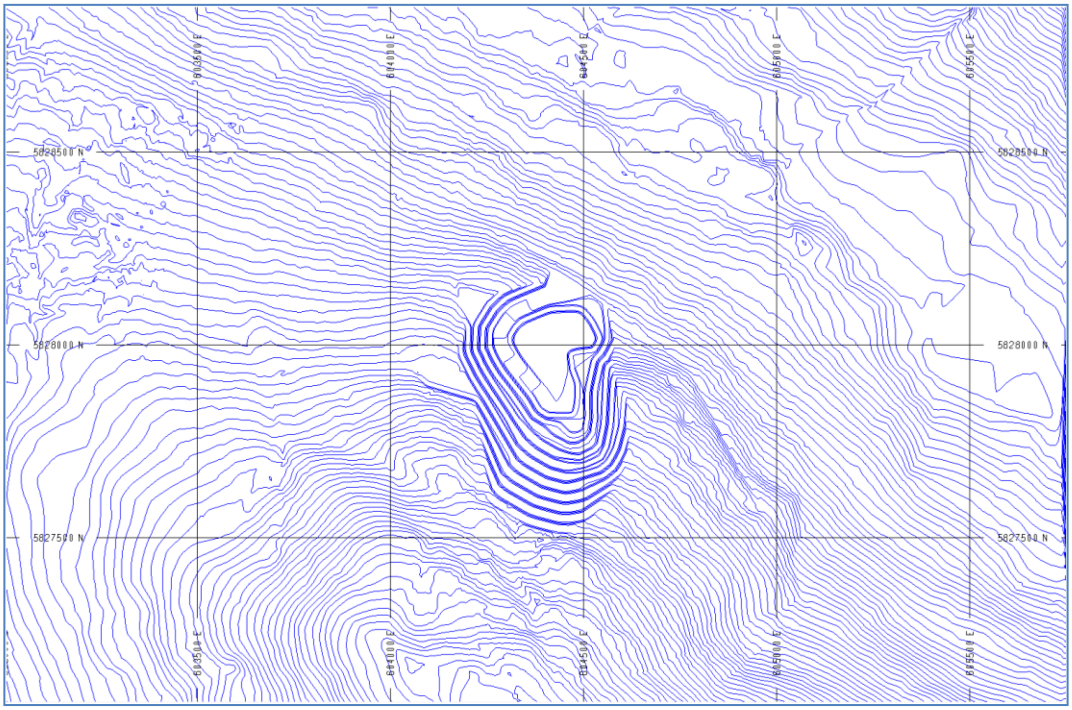
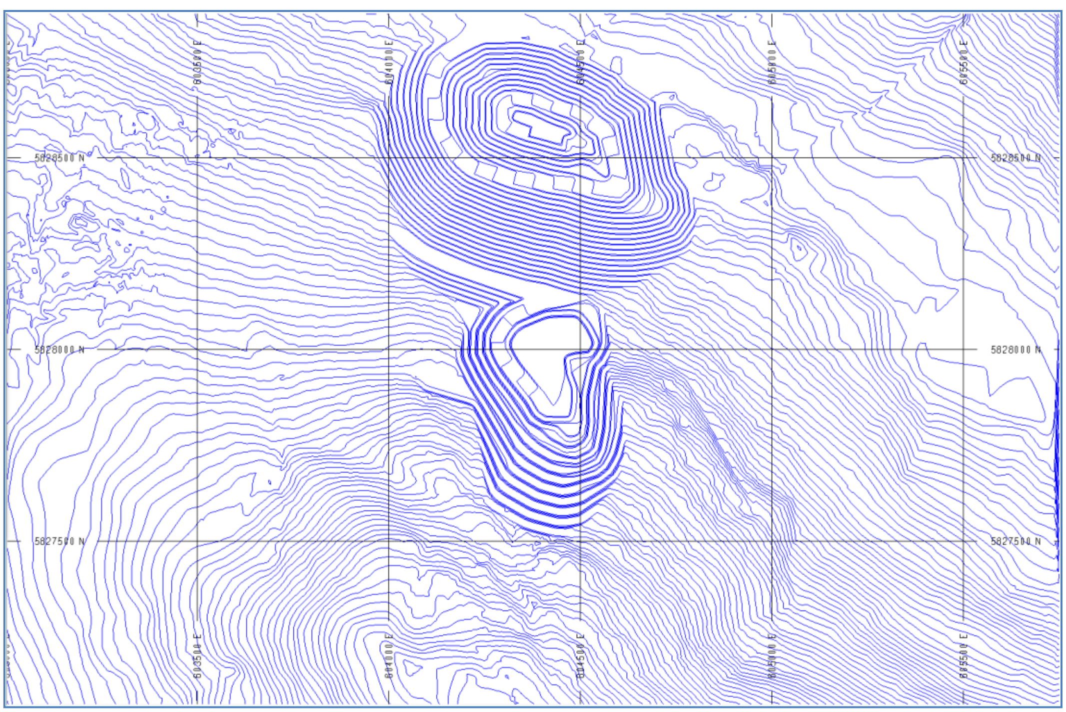


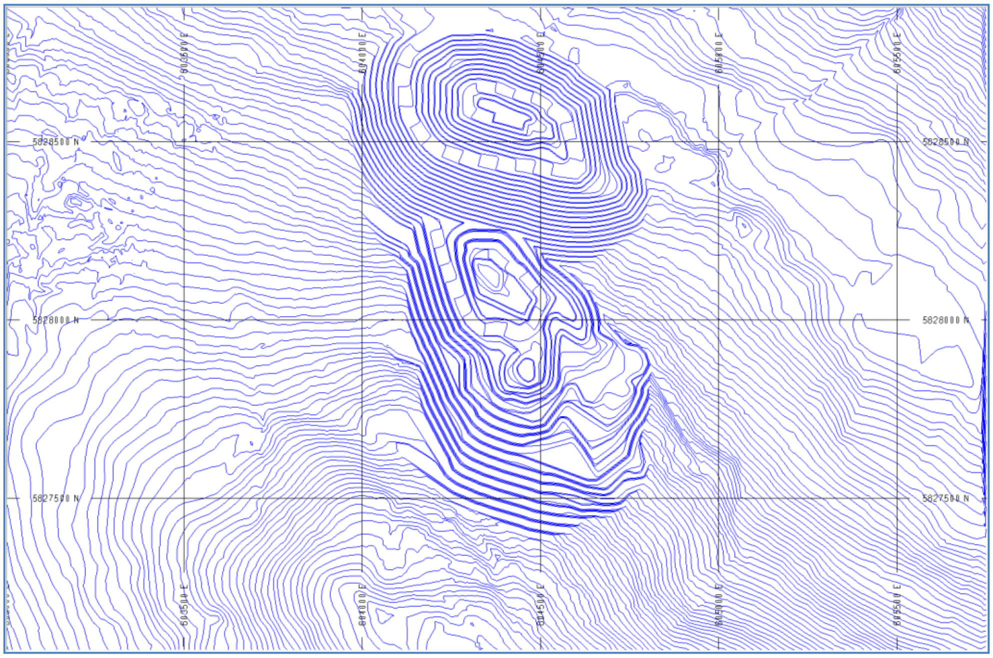
Figure 19-6: Phase 2 Design



Phase 3

The starting point for Phase 3 is further up the slope. Mining in this Phase is only in the Main Zone portion of the deposit, and targets a higher-grade zone. A ramp is left in the pit at the 1000 level. Figure 19-7 shows the pit with Phases 2 and 3 at the same time.

Figure 19-7: Phase 3 Design



Phase 4

Phase 4, the final phase, starts at the top of the mountain and uses the western side as access to the mine. Once it reaches the 1000 level, a new ramp system is developed parallel to the Phase 3 ramp system to go into the deposit. This is part of the pushback of the Phase. Figure 19-8 shows Phase 4 and the total pit at the end of the proposed mine life.

The final pit design closely approximates the LG shell developed for the 40,000 t/d case. The south slope ended up slightly flatter due to inclusion of the geotechnical berm. However, the wall slope near Spanish Creek closely followed the original design concept.

The final design is shown in solid green with the various benches and slopes in Figure 19-9. The 40,000 t/d pit shell is the dashed line. The thinner blue line represents the 25,000 t/d pit shell used for guidance on the North Zone pit wall near Spanish Creek.

The phase tonnages and grades were then used for scheduling at 40,000 t/d.

Figure 19-8: Phase 4 Design

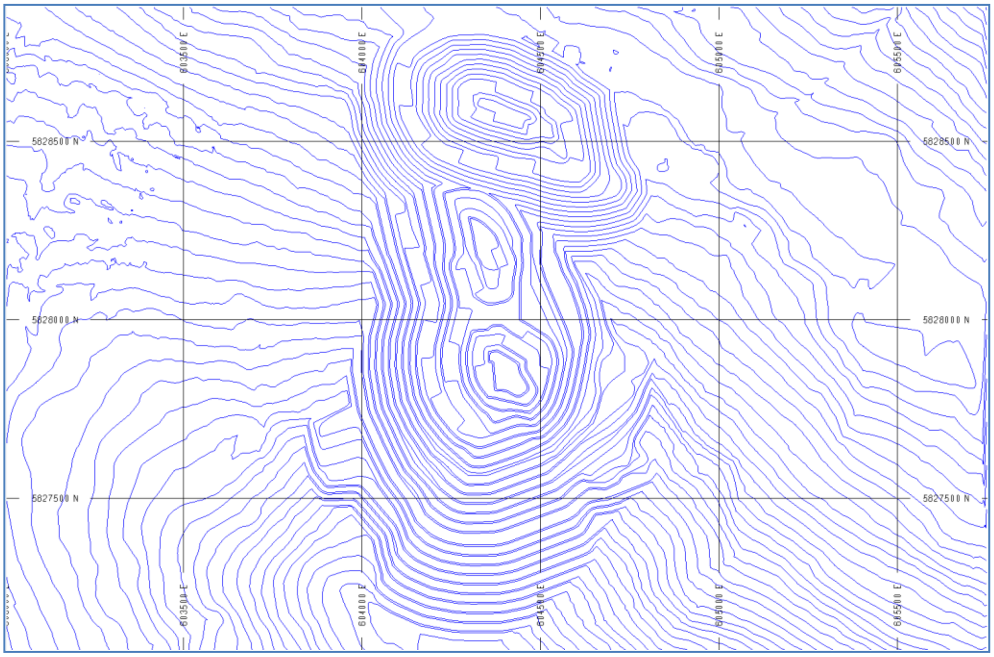
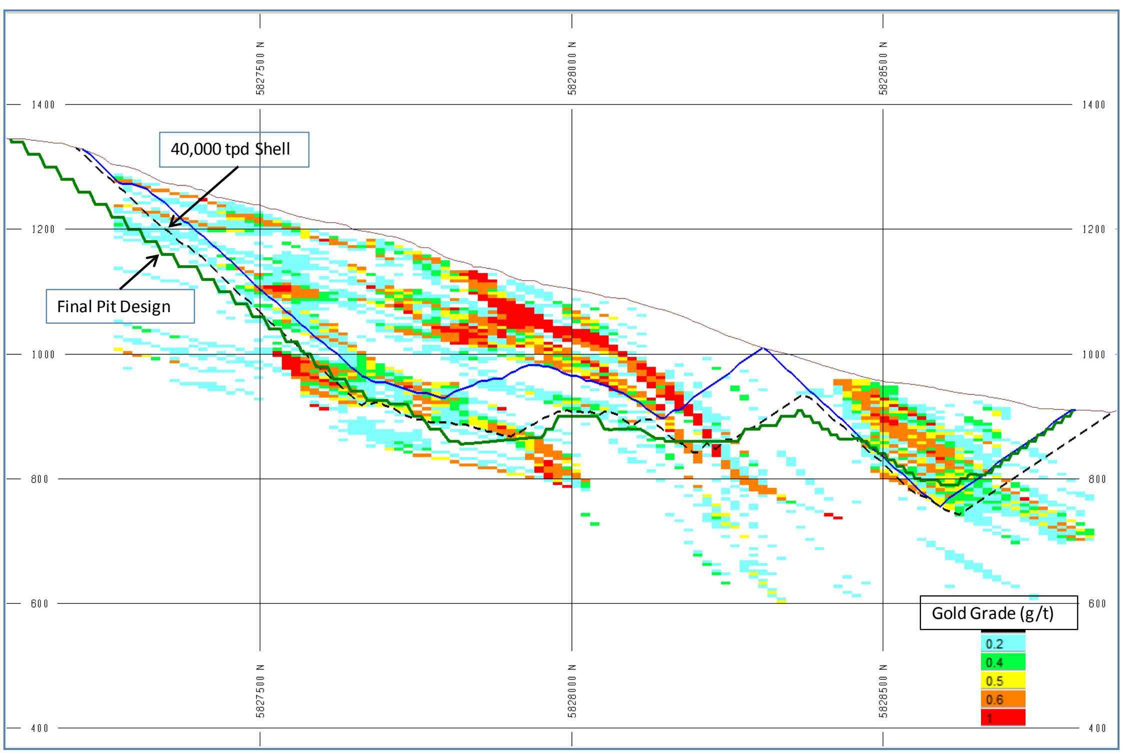


Figure 19-9: Final Pit Design – Cross-Section at Easting 604340



19.3.5 Mine Schedule

The plant feed material from the various phases was separated into low grade and high grade bins. The cutoffs for these bins were:

- Low Grade ....0.196 g/t < Low Grade < 0.400 g/t
- High Grade ....0.400 g/t < High Grade.

The low-grade cutoff is a milling cutoff based on the calculation using:

- Gold Price: ....US$950/oz
- Exchange Rate: ....C$1.10 = US$1.00
- Gold Recovery: 90%
- Gold Refining: ....US$8/oz
- Gold Payable: 98.5%
- Process Cost: ....$5.27/t feed
- G&A Cost: ....$0.52/t feed.

The results of the calculation was a milling cutoff of 0.196 g/t or 0.2 g/t.

During the trade-off study, a quick analysis had indicated that stockpiling of lower grade material, while it improved the grade for a short time, forced the mine to advance vertically too quickly to maintain plant feed at 40,000 t/d and did not provide a significant increase in the NPV. For the final schedule, no significant stockpiling of low grade or later processing was considered. Stockpiling of low-grade plant feed did occur as part of pre-stripping to position the mine to provide the 40,000 t/d of plant feed. The priority material reclaimed when the plant was in operation was the high grade, followed by the lower grade.

The schedule assumed that mining started in January of Year -2 with pre-stripping operations. NAG material was required for the tailings dam construction to reduce quarrying costs. Two schedules were developed with the following mining sequences, Phases 1, 2, 3, and 4, and Phases 1, 3, 2, and 4.

The second sequence provided the largest initial amount of NAG material for tailings construction. Phase 2 (North Zone Pit) has a higher percentage of waste material in the PAG classification, so shifting the timing of the Phase 2 mining benefited the tailings construction. The completion timing of Phase 2 was essentially the same as the first sequence, which still permitted backfilling in the final years of the mine.

The second sequence also provided a slightly improved plant feed grade profile in the initial years, which benefited the project economics. This is due to the lower grade present in

Phase 2. The strip ratio of Phase 2 is higher, so shifting its mining allowed for a lower overall material movement target over the life of the mine.

Plant feed was estimated to be 10.8 Mt in Year 1 to account for the ramp up of plant production. From Years 2 to 6, plant feed maintained at a rate of 40,000 t/d or 14.4 Mt/a. Year 7 forecasted a slight reduction in plant feed tonnage due to Phase 2 completion and Phase 4 being the primary source for feed. Years 8 through 10 showed declining tonnages, with Year 10 being only a partial year as the pit as designed is exhausted.

Waste movement is projected to peak in Years 2 through 5 with a maximum of 32.7 Mt. Years 6 to 8 maintain a level of 21 Mt. The final two years show dramatically reduced waste movement tonnages as the pit was near completion and the strip ratio declines. The completion of Phase 2 in Year 7 allows waste material from Phase 4 to be sent on a shorter haul to backfill Phase 2.

The plant feed tonnages by phase and grade over the life of the mine have been shown in Figure 19-10. Waste tonnages by phase are illustrated in Figure 19-11.

Detail on the mine schedule with the associated waste dump allocation has been included in Appendix D.

Figure 19-10: Plant Feed Tonnage by Phase with Total Feed Grade by Year

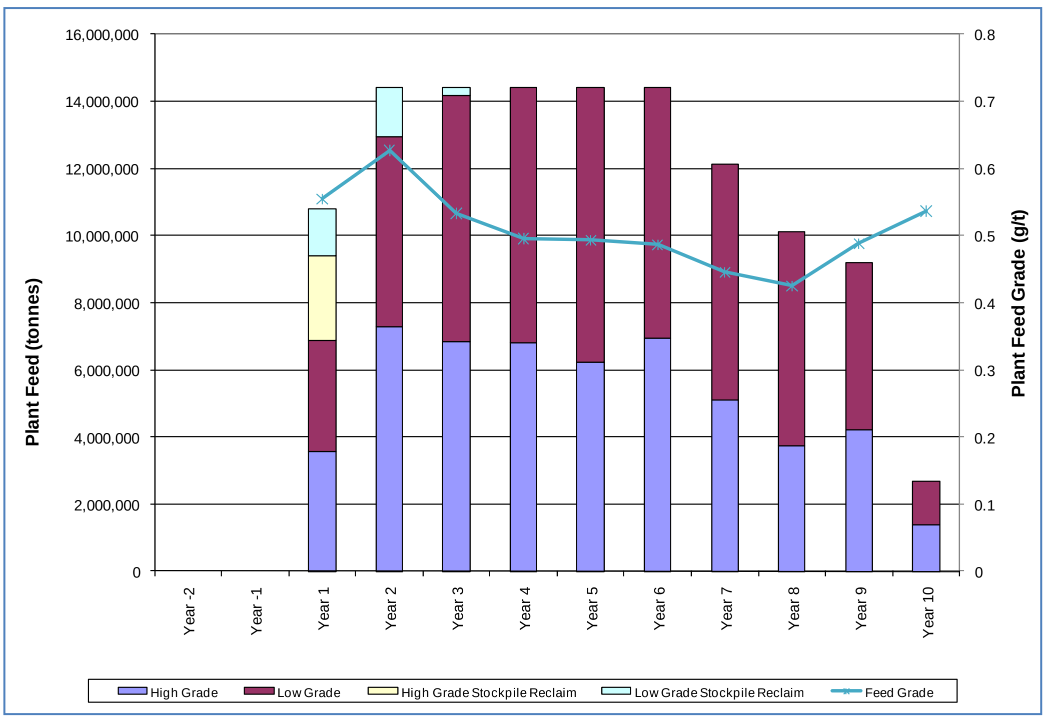
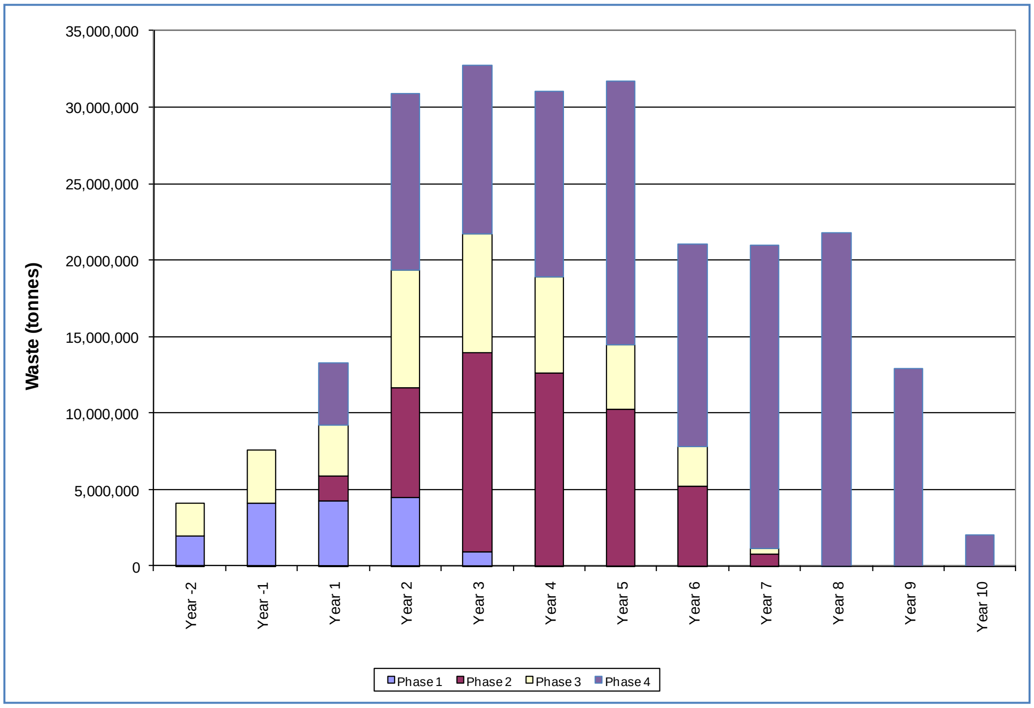


Figure 19-11: Waste Tonnage Mined by Phase and Year



19.3.6 Waste Dump Design

Waste material allocation followed the directions assumed previously.

These were:

- Acid Generating Material (AG) – to be stored in the tailings management facility (TMF)
- Potentially Acid Generating Material (PAG) – to be comingled with the NAG material, with the exception of 1.4 Mt annually sent to the TMF
- Non-Acid Generating Material (NAG) – to be initially used for construction of the TMF and additional TMF requirements over the mine life. The remainder is to be comingled with the PAG.

Four distinct waste dump locations were established and the waste materials stored were:

1. Tailings Management Facility (TMF)
   - Option H2 design from KP
   - Acid generating material, select PAG material, and NAG for construction

2. West Dumps (including Plant Dump)
   - NAG and PAG material on the western edges of the pit design
   - The plant dump material is stored adjacent to the process plant containing both NAG and PAG waste
3. East Dumps
   - NAG and PAG material stored on the eastern edges of the pit design
4. North Zone Pit Backfill
   - NAG and PAG stored in Phase 2 once mining of the phase is complete.

Waste volumes were determined using the specific gravity to determine bank volume then 30% swell applied. This was to calculate the loose volume for the waste dump required. Year-by-year allocation of waste by level was calculated including the north-west and southeast sections of the TMF. This detail has been included in Appendix D.

The volumes of waste in each dump have been shown in Table 19-6.

The Plant dump represents 45.9 Mlm^3 of the total West/Plant dump total of 66.6 Mlm^3.

The resulting waste dumps have been shown on the site layout plan. The waste dumps near the pit have been shown in an idealized overview presented in Figure 19-12.

During the mine life, drainage from the East and West Dumps will be directed to settling ponds near their bases. Drainage from these dumps at the end of the mine life will be directed back into the mined-out pit. The drainage from the Plant Dump, both during mine operation and at the close of mining, will be directed in ditches to the TMF to facilitate the final pit flooding in preparation for ultimately making a lake.

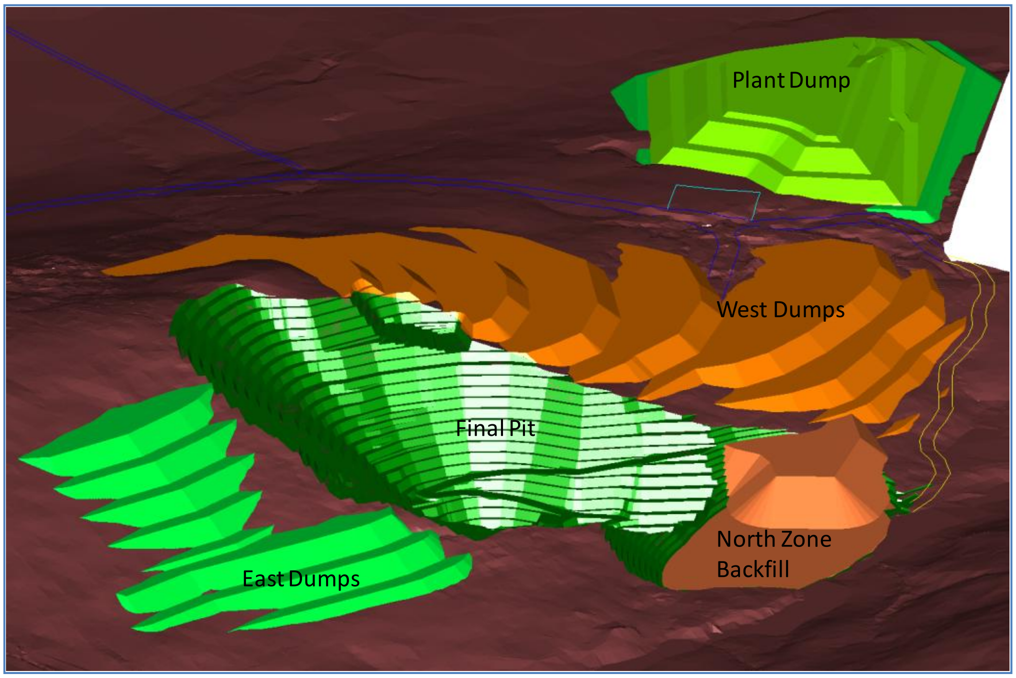
The North Zone backfill has been designed with a level portion at the same level as Spanish Creek. Opportunity exists to turn this into spawning grounds for fish in Spanish Lake by making a large gravel bed suitable for spawning just at the entrance to the lake.

All the dumps will have concurrent re-sloping and reclamation. The spacing on the lifts has been designed to allow easy re-sloping.

Table 19-6: Waste Volumes by Dump Location

Waste Storage Location	Unit	AG	PAG	NAG	Total
West Dump/Plant Dump	(lcm)	-	51,223,300	15,405,200	66,628,500
East Dump	(lcm)	-	2,119,500	890,200	3,009,700
TMF – AG/PAG	(lcm)	7,300	6,659,400	-	6,666,700
TMF – Northwest	(lcm)	-	-	3,722,100	3,722,100
TMF – Southeast	(lcm)	-	-	1,600,400	1,600,400
Total	**(lcm)**	**7,300**	**60,002,200**	**21,617,900**	**81,627,400**

Figure 19-12: Waste Dump Configuration



20 INFRASTRUCTURE

20.1 Infrastructure and Site Layout

The Spanish Mountain project infrastructure and site layout consists of the following:

- open pit
- waste rock dumps
- process plant, mobile equipment, and maintenance shops
- office/administration and dry complex
- tailings impoundment area.

20.2 Mine/Mill Site Operations

The Spanish Mountain mill is to be located to the south west of the open pit and west of the west end of Spanish Lake. Buildings containing offices, warehousing, welding shop, services shop and a mine equipment /maintenance shop will be in close proximity to the mill building. Refer to the illustration titled "Spanish Mountain Mine Site" for the plan in the Appendix E.

20.3 Mill Facility

The mill facility will consist of the processing facility and the supporting infrastructure for the mining operation as shown in the illustration "Spanish Mountain Mill Site" in Appendix E. This building contains the ore processing facility and mill services shop. A description of the milling and concentrating process can be found in the mineral processing and metallurgical testing section of this report.

20.4 Maintenance Facilities

A mobile equipment garage will be constructed with eight services bays, (one tire bay, one wash bay, and the remainder open pit truck maintenance bays) to accommodate the open pit and surface equipment. It will be required to maintain mine haulage trucks, service trucks, mobile mining equipment, service and maintenance vehicles, and personnel vehicles.

20.5 Warehouse, Office, and Dry

The main warehouse, office, and dry will be in one building location near the mill. The office section will accommodate administrative staff and warehouse personnel. The dry will accommodate all mine and process plant personnel.

20.6 Fuel Storage and Handling

The maximum fuel consumption will be 30,000 L/d and the fuel storage capacity at the site will be for five days of storage at 150,000 L. The tank farm is to consist of two 75,000 L storage tanks, with spill basins and containment.

20.7 Explosives Use and Storage

The explosives manufacturing facility occupies an area about 45 m x 35 m and is located approximately 500 m from the mill along the tailings access road. All explosives related structures will be located within an appropriately barricaded and fenced area in accordance with NRCAN Standards.

20.8 Roads

The site will provide roads connecting the open pit to the main processing area and to the service complex and tailings facility. The existing site access road from Likely will be upgraded and all roadways will not be paved but treated with lime.

20.9 Water Balance System

Approximately 6,700 m^3/d of fresh water will be required to satisfy water demand for the process plant. Water required for the operations will be pumped from nearby Spanish Lake into a water storage tank in the mill. Water will be drawn from this tank and pumped throughout the process plant and other locations where process water is needed. Tailings recycle water will make up the remainder of plant process water.

20.10 Service/Potable Water

Water required for the site services will be pumped from Spanish Lake into a water storage tank. Water will be drawn from this tank, treated, and pumped to the required locations where service water is needed. Potable water will be provided in bottled containers throughout the site.

20.11 Sewage Treatment

A sewage and wastewater treatment plant will be a self-contained Rotating Biological Contactor treatment plant, complete with clarifiers on both the inlet and the outlet, in addition to grease traps. The treated effluent will be discharged into the designated tailing pond during the construction period. When the mill begins operation the treated sewage effluent will discharge into the mill tailings system.

Wastewater collection will conform to all applicable regulations and good engineering practice. The collection system will consist of insulated HDPE pipe and shall not be connect to any source of industrial (mine) wastes. The piping will be installed on accurately graded granular material to ensure drainage or in culverts to allow unobstructed flow of traffic. Portions of the piping that cannot be positively drained will be heat traced.

20.12 Waste Management

Industrial and domestic waste from this site will be transported to local disposal sites.

20.13 Electrical and Backup Power

The anticipated power demand for the mill/mine complex is approximately 34 MW. This energy will be provided through a newly constructed transmission line from Gavin to site, and an upgraded line from Soda Creek to Gavin. There is the potential for the addition of Static VAR Compensation (SVC), which has not been included in the capital, in order to maintain voltage stability – this would have to be validated and costed through further study and analysis by the utility.

The main substation at the site will consist of two main transformers operating radially and connected to the main 15 kV switchgear. As a result, there will be two 15 kV bus arrangements sharing the load of the property; a tiebreaker will be available to tie the two arrangements together, but it is not intended that the transformers operate in parallel.

In order to provide back-up power to essential services, diesel generators have been allowed for in the mill and the office/dry complex.

A short section of aerial 15 kV power line will provide electrical power to some of the remote infrastructure.

21 MARKETS AND SMELTER

The process plant includes an electrowinning circuit for gold recovery. Initial metallurgical testwork has indicated that some silver may also be present in the final product but this has not been quantified for the PEA. It is recommended that further work be completed on this so that a value may be assigned.

A gold doré is produced from the process plant. For the purpose of this PEA level evaluation, AGP assumed the following refining terms based on present contracts:

- Refining Charge .... US$8/oz
- Payables... 99.5% of gold, 0% of any silver.

22 ENVIRONMENTAL

Project-specific environmental studies have been conducted since 2007, including aquatic resource studies (water quality and quantity, sediment quality), aquatic biota studies (fish species and community composition, fish habitat, primary and secondary productivity), terrestrial resource studies (wildlife and vegetation), and climatology.

First Nations and community engagement activities have been on-going since 2009. With the guidance of Catana Consulting and Knight Piesold, workshops have been held in 2010 with both the Williams Lake and X'atsull Indian Bands. Topics of these multiple workshops have included the basics of the mine life cycle, exploration, and the numerous on-going environmental studies being completed. Site visits by both First Nations communities have taken place and include Chief and band council members from Williams Lake Indian Band and elders from X'atsull.

.

22.1 Environmental Setting

22.1.1 Physical

The Spanish Mountain Gold Project is located in the Interior Cedar-Hemlock (ICH) and Engelmann Spruce-Subalpine Fir (ESSF) biogeoclimatic zones. The ICH zone occurs at low to mid elevations within the Project area and is characterized by cool, long, snowy winters and warm, dry summers. The ESSF zone is present within the higher elevations of the Project area and is characterized by cool, short growing seasons and long, cold winters.

The Project area spans the Cedar Creek and Spanish Creek watersheds. Cedar Creek discharges north and then west into Quesnel Lake, located at 729 masl. Quesnel Lake flows into the Quesnel River, which discharges to the Fraser River at Quesnel. The Cedar Creek watershed contains two small lakes, the lower Nina Lake at 972 masl, created in the mid-1930s by construction of the Cedar Lake Dam to store water for placer mining (Hartman and Miles, 2001), and Boswell Lake at 999 masl.

The Spanish Creek watershed, with a catchment area of 13,300 ha, is located northeast of Cedar Creek and drains north into the Cariboo River, which in turn discharges into the Quesnel River near Quesnel Forks. The Winkley Creek watershed, with a watershed area of approximately 500 ha, lies to the south of the Cedar Creek watershed and discharges directly into Quesnel Lake near Hobson Arm.

Five hydrology stations have been established in and around the Project area to record continuous water level data. Periodic discharge measurements will allow development of discharge rating curves in 2011 for input into the site-wide water balance.

Automated weather stations installed in the Project area and regional meteorology stations have been used to characterize the local and regional climate. Based on long-term precipitation data from the regional station at Barkerville, BC, the mean annual precipitation for the area is estimated as 1,014 mm. The average annual evaporation on site was estimated at 389 mm using evaporation information from the Mt. Polley Mine (located approximately 15 km east of Spanish Mountain) and adjusting the evaporation values based on elevation.

22.1.2 Chemical

Water quality monitoring sites have been established throughout the Project area to characterize existing water quality conditions. Results for water quality samples taken within the claim boundary have consistently shown concentrations of total and dissolved metals exceeding levels set by the Canadian Council of Ministers of the Environment (CCME) and the British Columbia Water Quality Guideline (BCWQG) limits for the protection of aquatic life, likely due to natural mineralogy of the claim area and disturbance from historic placer mining activities. Sample sites located outside of the claim boundary have shown no guideline exceedances, with a few exceptions (dissolved selenium and free cyanide at the Spanish Creek site, and dissolved and total selenium at the Winkley Creek site).

22.1.3 Biological

Provincial records indicate that bull trout, rainbow trout, and sockeye salmon are present within the Spanish Creek watershed, and rainbow trout are present in Cedar Creek. A series of falls and rapids in the lower reaches of Spanish Creek obstruct the upstream movement of anadromous fish. Bull trout, lake trout, and rainbow trout are the only species recorded in Spanish Lake.

Site-specific fish and fish habitat assessments conducted since 2007 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 in Cedar Creek, and juveniles Chinook were captured and adult Coho salmons were detected near the mouth of Spanish Creek.

Western red cedar and western hemlock are the dominant forest species in the ICH. Grizzly bears, black bears, caribou, bighorn sheep, and moose are common in the ICH zone. The dominant climax tree species in the ESSF zone are Engelmann spruce and subalpine fir.

Ungulates such as bighorn sheep and caribou, and furbearers such as fishers and wolverines are common in the ESSF zone. The Wells Grey herd of mountain caribou is located outside of the project area in the upper catchment of Black Bear Creek, approximately 15 km to the northeast. The range of the Quesnel Lake North population of grizzly bear covers the project area. The status is currently listed as viable for the Quesnel Lake North population.

22.2 Environmental Assessment and Permitting Process

A typical environmental assessment (EA) is generally completed within a two-three year period. During the pre-application phase, the proponent submits a Project Description to the BC Environmental Assessment Office (EAO) and the federal Canadian Environmental Assessment (CEA) Agency. The EAO and the CEA Agency assess whether the project will require an assessment based on any legislative triggers, and the formal scope of the review. The federal Major Projects Management Office (MPMO) then develops a Project Agreement among federal regulatory bodies that specifies how and within what timelines the Project will be reviewed. Detailed environmental and socioeconomic baseline studies are initiated following submission of the Project Description, and typically require a two-year period to complete. During this time, a draft Application Information Requirements (AIR) is prepared for both the provincial Application for an Environmental Assessment Certificate (Application) and the federal Comprehensive Study Report (CSR). Submission of the draft AIR is followed by regulatory meetings, at least one public open house, and ongoing First Nations consultation. The final AIR is issued incorporating relevant comments received during the review period.

Following completion of the baseline studies and based on Feasibility Studies, an impact assessment is prepared for the Application/CSR. Following submission of the draft Application/CSR to the EAO and federal agencies, a 30-day review period for completion and concordance with the AIR is initiated. The final Application/CSR is then submitted and the 180-day review period begins. The EAO completes an Assessment Report and Federal Responsible Authorities conclude their Notice of Decision within the 180-day period, based on detailed review of the Application/CSR and any comments received from First Nations or the public. Provincial Ministers complete their review and sign off on the Project within 45 days and the Provincial Environmental Assessment Certificate is issued. If Concurrent Permitting was conducted during the EA process, provincial bodies have 60 days to issue provincial permits. Federal Responsible Authorities then issue their Notice of Decision, and SMGL and consultants work with federal regulators to move forward on any required federal permits and authorizations.

The federal *Fisheries Act* prohibits the harmful alteration, disruption, or destruction of fish habitat without specific authorization. Construction of the TSF in the Nina Lake basin will 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 natural fish-bearing waters. The Department of Fisheries and Oceans Canada (DFO), Environment Canada (EC), and Natural Resources Canada (NRC) will conduct a thorough analysis of tailings management options, which includes public consultation, to ensure that the proposed use of the water body 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 the loss of fish habitat in Spanish Creek as a result of pit development, and in Cedar Creek as a result of reduced flows from diversion of surface runoff around the TSF.

22.3 Metal Leaching (ML)/Acid Rock Drainage (ARD) Characterization

Characterization of ML/ARD potential for the Spanish Mountain project has included preliminary testing of drill core composites from intervals chosen to represent waste rock, pit walls, and ore material. Four different lithologies were tested including argillite, conglomerate, greywacke, and siltstone. Prior to this characterization, no work had been completed to assess the ML/ARD potential of the deposit.

To characterize ML/ARD potential at Spanish Mountain, project geology was assessed with relevance to sulphide and carbonate mineralogy and acid-base accounting (ABA) tests were completed on drill core composite samples. Testing included the modified neutralization potential (NP) method (MEND 1991), in addition to paste pH, paste conductivity, total sulphur, sulphur as sulphate (sodium carbonate and hydrochloric acid methods), total barium, and total carbonate analysis. Elemental analyses of the samples were completed by aqua regia digestion and analyzed by ICP-MS.

The following AP/NP ratios were used to assess ARD potential from the Spanish Mountain deposit:

- NP/AP < 1 = potentially acid generating (PAG)
- NP/AP > 1 and < 3 = uncertain
- NP/AP > 3 = non-PAG.

Generally, NP/AP ratios below 1 indicate potential for ARD, whereas ratios above 2 indicate low potential for ARD. Ratios between 1 and 2 indicate uncertainty. However, due to the uncertainty in application of NP for this initial assessment, the threshold value of 3 was used for low potential for ARD. At low sulphur concentrations, interpretation of ARD potential using NP/AP ratios may not be meaningful because oxidation of small concentrations of sulphide produces low amounts of acid that are readily neutralized by many rock components in addition to carbonate. A sulphide concentration of 0.1% was nominally selected to represent low sulphur concentrations. Below this level, rock was classified as non-PAG regardless of the NP/AP. However, this criterion had no affect on ARD classification because NP/AP ratios were above 3 for all samples containing less than 0.1% sulphide.

Element scans provide an indication of the leaching potential of the rocks. Elements present at ten times typical global concentrations (Price, 1997) indicate potential for release under neutral pH conditions. Under acid generating conditions, metal mobility will increase regardless of metal concentrations in the rock.

Based on project geology, pyrite appears ubiquitous throughout the argillite units, ranging from 1% to 3% on average, with some instances of concentrations up to 35% (Peatfield, 2009). Iron carbonate alteration is present throughout the rock units, and noted to be most intense in the argillite units. Potential for ML/ARD appears to be highest in the argillite units, given the enrichment of pyrite in this rock type. Calculation of neutralization potential from carbonate concentrations will be overestimated due to the balance of alkalinity and acidity release from iron carbonates during oxidation and iron hydrolysis.

Results from static testing indicate that for all of the samples of rock core tested (n = 79), only four (5%) were classified as PAG, 36% as uncertain and 60% as non-PAG. By rock type, the most buffered are greywackes, followed by siltstone, conglomerate, and then argillite, which had the four PAG samples. Argillite NP/AP ratios ranged from 0.6 to 40, with an average of 1.7. Conglomerate NP/AP rations ranged from 1.5 to 79, with an average of 4.5. Greywacke ranged from 2.8 to 376, with an average of 9.4. Siltstone ranged from 1.8 to 449, with an average of 6.1. By mine material, NP/AP ratios for ore-grade samples ranged from 1.5 to 14, with an average of 3.2. Pit walls ranged from 2.0 to 228, with an average of 38. Waste rock ranged from 0.6 to 449, with an average of 29.

Results from element scans indicated that the concentrations of As, Ba, Co, Mo, Ni, Ag, and Se are greater than ten times global averages, an indication of potential leaching concerns for all sample types.

From this initial characterization program, it would appear that the Spanish Mountain resource has a low potential for ML/ARD, especially if waste segregation strategies can be incorporated into proposed mining methods. However, there are a proportion of uncertain PAG samples (36%) that may require management, especially considering the abundance of iron carbonates that do not contribute to NP. It should also be noted that the pit wall predictions would likely change as the mine plan develops.

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

Site-specific water quality modelling will evaluate the effects of any discharge to surface and ground water. 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.

22.4 Community and First Nations Engagement

Public comment in relation to the Project must be sought, addressed, and documented through public open houses, meetings and presentations, and through the provincial EAO Project Information Centre (e-PIC) and federal CEA registry.

The Spanish Mountain Project is located northeast of the community of Likely, BC, which has a population of approximately 350 people. Williams Lake is located 70 km southwest of the Project, and had a population of approximately 10,700 in 2006. Quesnel is located approximately 90 km northwest of the Project, and had a population of approximately 9,300 inhabitants during 2006. Other communities in the area include Horsefly, Black Creek, Keithley Creek, Quesnel Forks, and Big Lake.

The project will be in the traditional territories of the T'exelc (Williams Lake) and Xats'ull/Cmetem' (Soda Creek) First Nations, member nations of the Northern Secwepemc te Qelmucw (Northern Shuswap Tribal Society Council). The Statement of Intent Traditional Territory Boundary extends from the town of Clinton in the south to Valemount in the north and from Alexis Creek east to Clearwater.

Community and First Nations consultation has been initiated by SMGL and will be ongoing throughout the pre-application and review phases of the EA.

22.5 Reclamation and Closure

A mine closure and reclamation plan is required to ensure that developed areas are restored to viable and self-sustaining ecosystems, and that safety and end-use land objectives are met. A detailed closure plan will require more thorough studies that include an environmental evaluation of the mine wastes (dumps and tailings), ultimate pit wall compositions, hydrologic regimes, and end use. These studies are typically completed as part of the Feasibility Study. SMGL will provide financial assurance that reclamation can be completed through posting of a reclamation bond, as required by the Mines Act, and during operation will provide a closure plan every five years.

A preliminary estimate of the Year 5 reclamation bond has been completed and is described in Section 24.

23 TAXES AND ROYALTIES

This PEA study was completed pre-tax. No consideration for taxation has been included in the analysis.

As discussed in Section 4.3 of this PEA, there are two blocks of claims to which royalties are applied. The exact claims are highlighted in Table 4-1. The two claims pertain to:

1. Robert E. Mickle of Likely, BC
2. D.E. Wallster and J.P. McMillian (collectively known as the Underlyers).

The royalty payable to Robert E. Mickle has a provision for 2.5% NSR royalty payable to Mickle for any production from the twelve mineral claims to which he has an interest. A provision exists that 1% and 1.5% of this royalty may be purchased with the payment of $500,000 to Mickle. There is also a requirement in the agreement to spend an aggregate amount in the sum of $200,000 on the Mickle claims in the period of Years 6 to 10 of the agreement.

The agreement with the Underlyers, in addition to payment of shares and/or cash carries a 2% and 2.5% NSR royalty. This is payable to the Underlyers for any production on the CPW claim. One 1% may be purchased by payment of $500,000 to the Underlyers at the commencement of commercial production from the CPW claim.

The option to buy back the net smelter return royalty from both Mickle and the Underlyers has not been exercised by Spanish Mountain Gold. That decision will be made closer to a production decision as to the merits of that purchase.

For this study, these royalties have been excluded in the economic analysis.

24 CAPITAL AND OPERATING COSTS

24.1 Capital Costs

24.1.1 Summary

The capital costs for the Spanish Mountain Gold project are summarized in Table 24-1. The costs are based on the estimate for a 40,000 t/d processing plant using a standard floatation with Carbon in Leach circuit and gold electrowinning. The mine has a 10-year life with full production at 40,000 t/d for the first six years then tapering off until the mine is complete.

Table 24-1: Spanish Mountain Capital Cost Summary

Capital Category	Total Capital ($M)	Pre-Production Capital Year – 2 to Year -1 ($M)	Production Capital Year 1 ($M)	Sustaining Capital Year 2+ ($M)
Open Pit Mining	-	-	-	-
Processing	215.0	170.3	42.6	2.1
Infrastructure	87.4	77.1	1.2	9.1
Environmental	18.5	18.5	-	-
Indirects	70.4	57.4	9.5	3.5
Contingency	72.1	58.9	11.0	2.2
Total	**463.4**	**382.2**	**64.3**	**16.9**

Initial capital requirements (pre-production) as shown are $382.2 million. Production starts in Year 1 and the capital requirements may be partially offset by revenue in that year. Capital requirements for Year 1 total $64.3 million. The indirect and contingency values varied by capital cost item. The indirect and contingency values referred to in Table 24-1 are percentages of the direct capital numbers.

Table 24-2: Indirect and Contingency Percentages by Capital Category

Capital Category	Indirects (%)	Contingency (%)
Open Pit Mining	10.0	15.0
Processing	21.5	25.2
Infrastructure	27.6	18.1
Environmental	0.0	20.0

The percentages shown in Table 24-2 are calculated from various areas within each capital category. It is for this reason the percentages may not be an even number.

24.1.2 Mining

Mining capital for this analysis has been considered to be zero. A full lease of mine equipment is assumed and discussed in Section 24.2. Because the mining fleet is leased, indirects and contingencies have not been applied. If the mining fleet had not been leased, the capital cost would have totalled $85.7 million over the life of the mine.

24.1.3 Process Design and Metallurgy

The capital cost estimate for the processing plant is based on the base-case throughput of 25,000 t/d. The estimation method for this study is described below.

Basis for Estimation

Mechanical Equipment

- budget pricing was obtained for major mechanical equipment items. Minor items are based on recent (similar) projects
- major equipment masses were provided by vendors; smaller equipment masses (where not provided by vendor) were estimated using recent project data
- an erection rate for mechanical equipment was based on recent 2009-2010 costs
- transportation costs are either quoted, or estimated from source location, mass and/or size.

Civil Works

- a composite civil rate per square metre was determined for each area using rates derived from similar projects and studies in Canada
- mill building supply and erection rates were taken from recent projects.

Structural

- supply rates per tonne obtained from recent Canadian projects for various components of steel supply and erection. Rolled up into a rate per tonne for structural supply and structural erection
- structures mass estimated from similar projects and Spanish Mountain layouts
- transportation rate per tonne assumes local supply (BC).

Platework

- supply and erection rates per tonne obtained from previous projects for various components of platework supply and erection. Rolled up into one rate per tonne for platework
- vessel mass estimated from process design and compared to existing vessel database
- transportation rate per tonne assumes local supply (BC).

Piping

- factored on an area by area basis using as built database (previous recent projects)
- individual factors for piping supply, valves supply, piping erect, and valves erect.

Electrical and Instrumentation

- factored using the in-house database for gold and PGM concentrators.

Building Costs

- factored by area for specific building types using the in-house database for gold and PGM concentrators.

Transportation Costs

- Estimated using recent quotations for 20 ft and 40 ft containers as a basis. All equipment shipped trans-continentally is assumed to be packed in 40 ft containers. Transportation cost per item assumes a percentage of container volume, and thus cost.

Capital Cost Estimate

In order to develop a capital cost for the 40,000 t/d scenario, the base-case capital estimate was scaled according to the widely accepted method of the six-tenths rule, specifically:

$$Capital\ Cost\ B = Capital\ Cost\ A\ \times (\frac{plant\ throughput\ B}{plant\ throughput\ A})^{6/10}$$

The process plant capital cost estimate at 40,000 t/d is summarized in Table 23-3. Total direct costs for the plant are estimated at $212.9 million. Indirect costs, including site establishment, commissioning, reagents and consumables, and the EPCM contract are estimated to add an additional $45.8 million. Contingency was estimated at $53.7 million for a total capital cost of $312.4 million.

Table 24-3: Capital Cost Summary for the 40,000 t/d Processing Plant

Plant Capital	Total ($)
Process Plant	
Civil and Earthworks	22,979,117
Mechanical	99,079,583
Structural	11,981,132
Platework	7,440,461
Piping	15,416,631
E&I	21,421,365
Building	34,573,985
Sustaining Capital	2,128,923
Total Process Plant Directs	**215,021,190**
Contingency	53,701,550
Pre-Project commitments	
Construction Site Establishment	6,386,768
Commissioning Costs (1 month, labour & spares)	721,000
On-site Vendor Commissioning	935,708
Consumables & Spares	
First Fill Balls/Reagents/Lube	3,724,259
Mechanical & Electrical Maintenance Spares	3,742,832
EPCM + Owners Costs	
EPCM Contract	27,675,994
External Services and Consultants	450,000
Total Indirects	**43,6363,560**
Total	**312,359,300**

24.1.4 Infrastructure – Site Layout

Infrastructure capital costs, required by the mining and milling operations, are listed in Table 24-4. These costs are based on information from published data and from previous work on similar operations.

To the site infrastructure direct capital costs an indirect percentage of 20% was applied and a contingency of 15%.

The mine/mill facility costing was based on pricing from similar projects. Vendor pricing were solicited for items such as the water pumping and piping, fuel storage and dispensing, and water treatment plant.

Table 24-4: Site Layout Costs

Infrastructure Capital	Total Capital Cost ($)	Capital Year -2 and Year -1 ($)	Capital $ Year 1
Power – Power Line Upgrade	8,000,000	8,000,000	-
Power – Electrical Substations	13,600,000	13,600,000	-
Power – Pit Power Lines	400,000	400,000	-
Explosives Storage Area	300,000	300,000	-
Haul Road Construction	3,500,000	3,500,000	-
Fuel Storage	300,000	300,000	-
Shop and Garage	8,650,000	8,650,000	-
Fresh Water and Pumping System	1,800,000	1,440,000	360,000
Mobile Equipment	500,000	500,000	-
Communications	150,000	150,000	-
Office	3,500,000	2,625,000	875,000
Access Road to Plant	1,750,000	1,750,000	-
Owners Cost	8,000,000	8,000,000	-
Sub-Total Capital Cost	50,450,000	49,215,000	1,235,000
Indirects (20%)	10,090,000	9,843,000	247,000
Contingency (15%)	7,567,000	7,381,750	185,250
Total Infrastructure – Site Layout	**68,107,000**	**66,440,000**	**1,667,000**

The electrical power line upgrade was estimated with 25 km of power line upgraded at $200,000/km and 20 km of new power line from the Gavin Substation at $150,000/km. The electrical substation estimate was created in detail for a 25,000 t/d plant then factored up with the 6/10^{ths} rule for use in the final 40,000 t/d estimate. The mine power line cost uses a cost of $100,000/km for 4 km of pit power line construction. The estimate allows for electrical transmission and distribution infrastructure as well as site communications. Several assumptions have been made in the costing of this portion of the project.

Initial haul road construction costs were estimated using a unit cost of $500,000/km for the 33 m wide road requirement. A total of 7 km will be required for access to the pit, North and Main Zone, as well as to the TMF. The access road to the plant will be for light vehicles and as such, the cost is lower at $1.75 million.

The shop and garage estimate covers the cost necessary for building eight bay shops. This will include a tire bay, wash bay, welding bay and electrical bay in addition to four service bays for the equipment fleet.

No construction camp is developed due to the proximity of the site to local towns. Construction trailers have been considered as part of the indirect costs for Spanish Mountain Gold personnel until the office is complete. The construction contractors would provide their own construction trailers.

A cost of $8.0 million has been included in the infrastructure area to cover such areas as surface rights acquisitions, Owner's project development team and any item associated with the Project that would be carried by Spanish Mountain Gold that would not be considered as sunk costs. The Owner's cost will be refined in planned future mining technical and environmental studies.

24.1.5 Infrastructure – Tailings Management Facility

An initial and sustaining capital cost estimate was completed for the following components of the TMF:

- Site preparation – TMF
  - including logging, service road construction, pipeline corridor construction, construction dewatering, and sediment and erosion control using Best Management Practices (BMPs)
- Earthworks and Foundation preparation for both TMF embankments
  - the total earthworks costs are integrated between the KP estimate and AGP's mining estimate, with AGP covering much of the material haulage costs from the open pit
  - seepage recovery and recycle systems
- Diversion ditch extensions
  - diversion ditches along roads included in road construction costs and Boswell Lake overflow channel
- Electrical
  - electrical infrastructure for seepage recovery and recycle systems, tailings pipelines and reclaim pipelines and the reclaim barge
- Mechanical
  - tailings pipelines and off-take valves
  - reclaim pipelines and reclaim barge
- Monitoring and Instrumentation
  - groundwater monitoring/seepage collection wells
  - TMF embankment monitoring instrumentation including inclinometers and vibrating wire piezometers
- EPCM – Engineering procurement and construction management

- Mobilization and Demobilization
- Indirects
- Contingency.

Detail on the TMF costs have been shown in Table 24-5. These have been summarized into different categories

The TMF cost estimate was prepared to an accuracy of ±40%.

Development of initial and sustaining capital costs for the TMF necessitated assumptions of the geotechnical site conditions, which must be verified.

The cost estimate was compiled using information from similar projects, engineering experience, and built-up unit rates based on standard contractor rates in British Columbia.

Table 24-5: Infrastructure – TMF Capital

TMF Infrastructure - Capital	Total Cost ($)	Year -2 and Year -1 ($)	Year 1 ($)	Year 2+ ($)
Direct Capital				
Site Preparation	2,554,000	1,935,000	-	619,000
Waste Management Facilities	14,580,000	7,922,000	-	6,658,000
Diversion Ditches	575,000	575,000	-	-
Electrical	1,853,000	1,853,000	-	-
Mechanical	16,486,000	15,089,000	-	1,397,000
Monitoring and Instrumentation	950,000	490,000	-	460,000
Total Direct Capital	**36,997,000**	**27,863,000**	**-**	**9,134,000**
Indirects				
EPCM (10%)	3,700,000	2,786,000	-	914,000
Mobilization/Demobilization (8%)	2,960,000	2,229,000	-	731,000
Indirects (20%)	7,399,000	5,573,000	-	1,826,000
Total Indirects	14,059,000	10,588,000	-	3,471,000
Contingency				
Earthworks (25%)	4,427,000	2,608,000	-	1,819,000
Electrical, Mechanical, Instrumentation (20%)	3,858,000	3,486,000	-	372,000
Total Contingency	**8,285,000**	**6,094,000**	**-**	**2,191,000**
Total TMF Capital Cost	**59,341,000**	**44,545,000**	**-**	**14,796,000**

24.1.6 Environmental

Fisheries Compensation – Tailings Management Facility

Habitat compensation costs for the TMF were developed assuming that any fish habitat lost or altered as a result of mine development will be replaced as per Department of Fisheries and Oceans (DFO) policy. Both direct footprint impacts and indirect downstream flow impacts were considered potential harmful alteration in the assessment. Instream compensation areas were calculated based on an estimated mean channel width of 5 m for fish-bearing main stem channels and 3 m for fish-bearing tributary channels, and riparian compensation areas were calculated based on 30 m setback widths for main stem channels and 15 m setback widths for tributaries. The compensation areas of main stem channels downstream of the TMF that would be harmfully altered as a result of reduced flows were also calculated based on an estimated mean channel width of 5 m.

Development of the TMF will directly affect Nina Lake, the main stem of Cedar Creek, and several unnamed tributaries. Fish habitat compensation ratios were calculated as 1:1, with assumed unit area capital costs of $150,000 for instream habitat and $50,000/ha for riparian habitat.

The estimated cost for fisheries compensation was $10 million payable in Year -2 upon commencement of construction of the TMF.

Reclamation Bond

A cost estimate for the Year 5 reclamation bond was prepared for the TMF (including the tailings and reclaim pipelines), plant site, and waste dumps. The open pit is assumed to be flooded following mine closure, and any additional bonding related to the open pit was not considered.

The closure and reclamation bond cost assumes the following:

- all equipment will be decommissioned and removed
- TMF embankment and beaches and the plant site area will be revegetated
- the waste dumps will be resloped concurrent with mining and revegetated
- water from the TMF will not require treatment post-closure.

The cost estimate was compiled using information from similar projects. This bond was estimated at $8.5 million payable in Year -2, once construction commenced on the TMF and site.

24.1.7 Indirects

The indirect costs as has been noted vary between capital cost categories. Table 24-6 shows the percentages that were applied to each capital cost category. This is to account for various items including construction supervision, erection of equipment, first fills, construction offices, and others.

Table 24-6: Indirect Percentages Applied

Capital Category	Indirects (%)
Open Pit Mining	10.0
Processing	21.5
Infrastructure	27.6
Environmental	0.0

The infrastructure percentage is a blended percentage as a result of the various items highlighted in the detailed tables.

24.1.8 Contingency

Contingency costs have been estimated based on various percentages applied to the direct capital costs. Table 24-7 shows the percentages and illustrates the level of confidence in each of the direct capital cost estimates.

Table 24-7: Contingency Percentages by Capital Cost Category

Capital Category	Contingency (%)
Open Pit Mining	15.0
Processing	25.2
Infrastructure	18.1
Environmental	20.0

24.2 Operating Costs

24.2.1 Summary

Operating cost development is for a 40,000 t/d mining and milling operation running for 10 years. This production rate was chosen in a trade-off study because it offered improved economics over lower production rates with anticipated higher gold prices and the known resource. A single open pit is mined in four phases with waste material placed adjacent to the pit, near the plant, backfilled in Phase 2 and PAG material hauled to the TMF and stored subaqueously.

All prices in this PEA are quoted in 3Q 2010 Canadian dollars unless otherwise noted. Where an exchange rate to American dollars is applied, a rate of C$1.10: US$1 is considered. Diesel fuel is assumed to cost $0.73/L and electricity costs $0.04/kWh.

The open pit is developed using conventional rotary drilling, blasting and loading with hydraulic shovels and 180-tonne trucks. The drills will be diesel powered to facilitate movement within the pits, while the hydraulic shovels will be electric powered to reduce operating costs. The open pit mine will have a LOM strip ratio of 1.97:1. A total of 116.91 Mt of material will be supplied to the mill from the open pit, while 230.1 Mt of waste will be moved.

Waste material has been broken into three classifications: AG, PAG, and NAG with respect to the acid generating potential. A total of 16,000 tonnes of waste material is expected to be acid generating and would be stored in the TMF. The PAG material represents 72% of the total waste material but is borderline with respect to its acid generating capability. An assumption that 14.2 Mt of PAG (6%) would be of questionable quality and would also be stored in the TMF. This is accounted for in the haulage costs and the size of the TMF to ensure that upon closure, the material would be stored subaqueously. The NAG material is the remaining 28% of the waste material. This will be used in the construction of the TMF in addition to being comingled with the PAG material due to its high buffering.

Phase 2 or the North Zone pit will be backfilled when mining is complete in that Phase in Year 7. Only PAG and NAG material will be stored there. Any AG or PAG with concerns will be stored in the TMF. A total of 45.6 Mt will be backfilled or approximately 20% of the total waste material.

Mill feed material will be mined starting in Year -2 during the pre-stripping of the mine. This will be stored adjacent to the primary crusher location. It will reach a maximum tonnage of 5.6 Mt prior to plant production commencing. Waste during this period will be stored in the TMF footprint; NAG in the embankments and PAG lining the base of the TMF. A small amount of NAG material will be used to build access roads around the pit area and on the site. Mining will commence at 40,000 t/d in Year 1 with a ramp up period and continue at that rate until Year 7 when production will start to taper off as the mining occurs in a single phase. Mining will be completed in Year 10. The stockpiled mill material will be drawn down by the end of Year 2.

The process plant is designed to operate at a nominal tonnage of 40,000 t/d with feed material from the mine and initially from the stockpile. The plant will use conventional grinding and flotation, with a CIP circuit and electrowinning to make a gold doré. Tailings will be pumped downhill to the TMF a distance of 2.7 km from the plant.

G&A costs are based on 13 salaried staff and 31 hourly personnel. Employees will be located in the immediate area and no camp is planned or required.

Table 24-8 shows a summary of all operating cost categories on a cost per tonne mill feed basis and a cost per recovered gold ounce.

Table 24-8: Operating Costs Summary

Cost Centre	Total Operating Cost ($M)	Cost Per Tonne ($/t mill feed)	Cost per Ounce (US$/oz)
Open Pit – Mill Feed and Waste	437.9	3.75	231
Leasing Cost	99.6	0.85	53
Processing + Tailings	598.1	5.12	315
G&A	49.9	0.43	26
Total	**1,185.4**	**10.15**	**625**

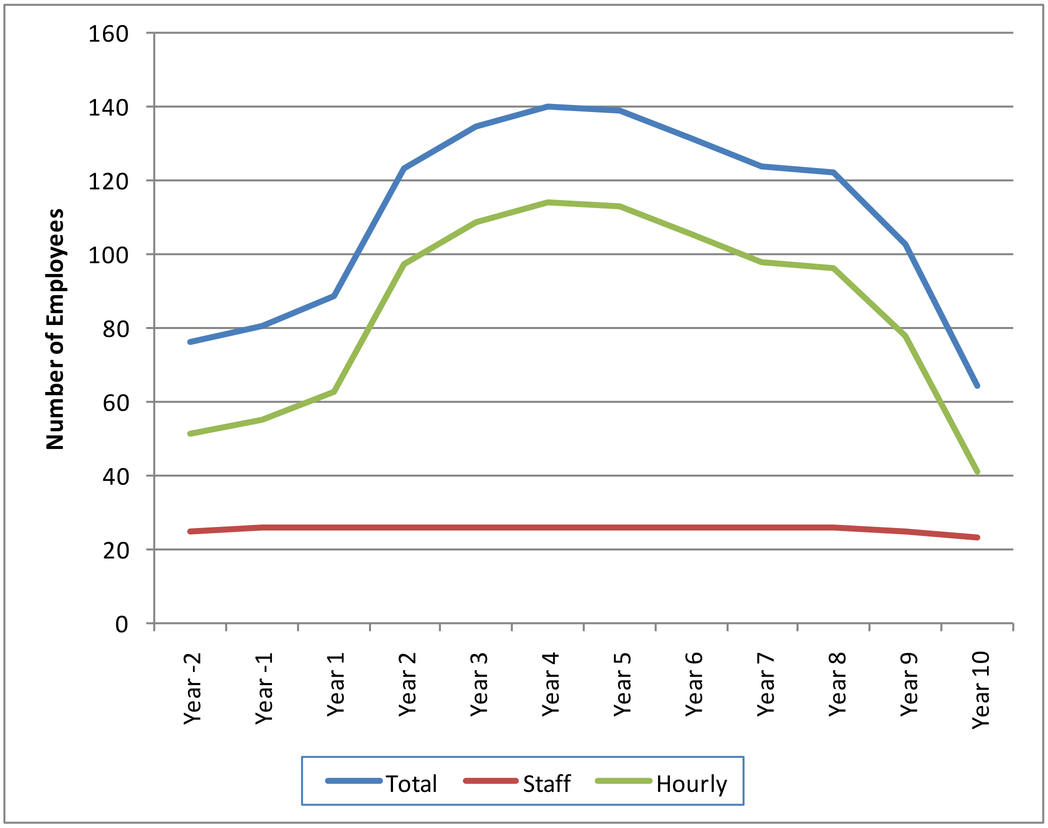
24.2.2 Mining

Mine operating costs were developed from base principles using hourly rates provided by vendors present in the area. These hourly rates were based on an owner-operated scenario with the vendor providing direct technical support in maintenance and training.

Key inputs into the mine operating cost estimate fuel, electricity, and labour. The diesel fuel cost for this study was estimated at $0.73/L. Electricity was set at $0.04/kWh.

Labour costs were developed jointly between Spanish Mountain Gold and AGP using collective agreements from nearby Mt. Polley Mine and Gibraltar Mine. Burdens were calculated to average 23% for Staff and 25% for Hourly personnel. Mine shifts were assumed to be using a 12-hour shift schedule. Mine operations labour requirements for the 40,000 t/d mine have been shown graphically in Figure 24-1.

Figure 24-1: Open Pit Labour Requirements



Maintenance staff numbers are reduced because the majority of the work is completed by the vendor on the equipment.

Open pit mining at Spanish Mountain Project is designed to utilize proven technology and equipment. Rock drilling is accomplished with the use of 200 mm rotary blasthole drills. These drills are diesel powered to provide for greater mobility within the pit. Rock haulage is handled using the 180 tonne class of trucks. Selective mining is possible with the use of electric hydraulic shovels. Also due to the short mine life, the lower capital costs relative to the cost of a traditional cable shovel also played a role in the selection process. Track dozers, graders, and rubber-tired dozers round out the major equipment list. Support equipment includes water trucks, a small backhoe with rock hammer, utility loaders, blasting loaders, pickup trucks, small submersible pumps, and light plants. Table 24-9 shows the equipment requirements.

The mine equipment requirements remain fairly constant until Year 7 when they drop due to backfilling of Phase 2. The trucks were required earlier and these remain in the fleet but will be utilized as required.

Table 24-9: Mine Equipment Requirements

Equipment	Capacity	Year -2	Year -1	Year 1 to 10
Production Drill	200 mm	1	1	2
Front-end Loader	21 m^3	1	1	1
Hydraulic Shovel	21 m^3	-	1	2
Breaker Loader	6.5 m^3	1	1	1
Haulage Truck	180 tonne	4	5	13
Tracked Dozer	306 kW	2	2	3
Grader	233 kW	2	2	2
Rubber Tired Dozer	350 kW	1	1	1
Backhoe and Hammer	2.3 m^3	1	1	1
Water Truck	-	2	2	2
Tool Carrier	-	1	1	1
Blasting Loader	-	2	2	2
Light Plants	-	7	7	7
Crew cab Pickup Trucks	-	2	2	2
Blasters Truck	-	1	1	1
Pumps	-	2	2	2
Pickup Truck	-	2	2	2
Manbus	-	1	1	1
Ambulance	-	1	1	1
Fire Truck	-	1	1	1
Lowboy	-	1	1	1

The large front-end loader pioneers initial development of the phases in the initial years. As the mine matures, the loader is responsible for 20% of both mill feed and waste material loading. The hydraulic shovels are responsible for the remainder. The small front-end loader would be used at the primary crusher tramming material from temporary piles to ensure the primary crusher is properly charged. Additional jobs would include general work around the crusher and TMF as required, in addition to snow removal.

As part of the operating cost, mine general and engineering is included. This covers the mine operations supervision and support staff, mine engineering and geology cost functions.

Drilling for the open pits is completed using two diesel drills with a 200 mm diameter bit in a rotary configuration. The pattern size used varies for mill feed and waste and is shown in Table 24-10.

Table 24-10: Drill Pattern Specification

Equipment	Unit	Mill Feed	Waste
Bench Height	m	10.0	10.0
Sub-drill	m	2.2	2.3
Blasthole Diameter	mm	200	200
Pattern Spacing – Staggered	m	8.3	8.6
Pattern Burden – Staggered	m	7.2	7.5
Hole Depth	m	12.2	12.3

The wider pattern spacing was considered possible due to the weaker nature of the rock being mined. The greater sub-drill was included to allow for caving of weaker zones without having to redrill the hole.

Table 24-11 shows the parameters used to estimate drill productivity.

Table 24-11: Drill Productivity Criteria

Drill Activity	Unit	Mill Feed	Waste
Pure Penetration Rate	m/min	0.49	0.49
Hole Depth	m	12.2	12.3
Drill Time	min	25.15	25.36
Move, Spot and Collar Blasthole	min	3.00	3.00
Level Drill	min	0.25	0.25
Add Steel	min	-	-
Pull Drill Rods	min	0.50	0.50
Total Setup/Breakdown Time	**min**	**3.75**	**3.75**
Total Drill Time per Hole	**min**	**28.9**	**29.1**
Drill Productivity	**m/h**	**25.3**	**25.4**

Local vendors provided explosive costing representative of the current operations in the area. A heavy ANFO product was specified and used in the costing of the explosives. The vendor also provided costing for delivering the product to the hole. This meant that Spanish Mountain Gold would be responsible for the blasting crew to charge the holes. The powder factors used in the explosive calculations are shown in Table 24-12.

Table 24-12: Design Powder Factor

	Unit	Mill Feed	Waste
Powder Factor	kg/m^3	0.62	0.56
Powder Factor	kg/t	0.22	0.22

Loading costs were estimated using the hydraulic shovels as the primary material movers. The front-end loader in the first two years (pre-stripping) would be responsible for phase initiation and the bulk of material mined. The first shovel will start in Year-1 with the second shovel starting the following year. When both shovels are functional, the loader will mine 20% of mill feed and 20% of waste on an annual basis. The loading percentage averages and other loading information are shown in Table 24-13.

Table 24-13: Loading Parameters

Mining Equipment	Units	Front-End Loader	Hydraulic Shovel
Waste Tonnage Loaded	%	20	80
Mill Feed Tonnage Mined	%	20	80
Bucket Fill Factor	%	95	95
Cycle Time	sec	35	30
Trucks Present at the Loading Unit	%	80	80
Loading time	min	3.62	3.20

The trucks present at the loading unit refers to the percentage of time that a truck is available to be loaded. To maximize truck productivity and reduce operating cost, it is more efficient to slightly under-truck the shovel. The single largest operating cost item is the haulage and minimizing this cost by maximizing truck productivity is crucial to lower operating costs. The value of 80% comes from typical standby a shovel encounters due to a lack of trucks.

Haulage profiles were determined for each pit phase for plant, waste dump, or TMF locations. From these profiles, Caterpillar's FPC software was used to determine haulage cycle times. These cycle times were applied to the appropriate yearly tonnage by destination and phase to estimate the haulage costs.

Support equipment costs were determined using either a percentage applied to the truck hours or the loading hours. As indicated earlier, these percentages resulted in the need for three track dozers, two graders and one rubber-tired dozer. Their tasks include cleanup of the shovel face, roads, dumps, and blast patterns. The graders will maintain the plant feed and waste haul routes.

The equipment rates applied, less the operating labour, are shown in Table 24-14. All rates include consumables such as fuel, tires, drill steel, bits, as well as associated maintenance costs from the vendor. Fuel consumption is estimated from base principles using the FPC software as a check. Operating labour is calculated separately.

Table 24-14: Major Equipment Hourly Rates

Equipment	Hourly Rate ($/h)
Production Drill	327
Front-end Loader	424
Hydraulic Shovel	398
Breaker Loader	114
Haulage Truck	285
Tracked Dozer	162
Grader	94
Rubber Tired Dozer	101

The mining cost is calculated by year to take into account changing haulage routes which helps in determining equipment requirements. To the mine operating cost was added the sampling cost. Every blast hole was assumed sampled to help identify gold boundaries, as well as provide an assessment of the material waste classification. A sample cost of $40 per sample was applied.

Detail on the mining cost has been included in Appendix D. The LOM average cost is shown in Table 24-15. This cost is for the total material moved. Leasing costs have not been included in this calculation.

Table 24-15: Open Pit Mine Operating Unit Costs

Open Pit Operating Category	LOM Cost ($/t total material)
General Mine and Engineering	0.12
Drilling	0.08
Blasting	0.11
Loading	0.18
Hauling	0.58
Support	0.18
Sampling	0.01
Total	**1.26**

24.2.3 Lease Cost

Given the higher capital requirements of the mine, the decision to lease the mining equipment was taken after other recent projects indicated that they were also applying the concept. The lease costs were developed for all pieces of equipment and continue for the life of mine. An amortization period of 60 months was applied to major equipment with a 24-month lease on pickup trucks. The costs are accumulated on an annual basis and the cost per tonne of mill feed determined. The life-of-mine cost of leasing amounts to $99.6 million or $0.85/t of mill feed. If the equipment were purchased outright, the capital cost for the mining fleet would have been $85.7 million. The differential is based on the 3% lease rate applied.

Table 24-16: Leased Open Pit Mining Equipment

Equipment	Lease Cost ($/Unit)	Operating Life (h)	LOM Fleet Cost ($)
Production Drill	1,739,000	25,000	6,956,000
Front-end Loader	5,043,000	35,000	5,043,000
Hydraulic Shovel	9,158,000	60,000	18,316,000
Breaker Loader	1,159,000	20,000	2,319,000
Haulage Truck	3,710,000	60,000	48,226,000
Tracked Dozer	1,275,000	35,000	6,376,000
Grader	927,000	20,000	3,710,000
Rubber Tired Dozer	1,507,000	30,000	3,014,000
Utility Backhoe with hammer	585,000	10 years	585,000
Water Truck (Sterling)	336,000	10 years	672,000
Tool Carrier	406,000	10 years	406,000
Blasting Skid Steer Loader	69,000	5 years	276,000
Light Plants	18,000	4 years	379,000
Lube/Fuel Truck	359,000	10 years	359,000
Mechanics Truck	244,000	4 years	-
Welding Truck	233,000	6 years	-
Crewcab Pickups	55,000	2 years	662,000
Blasters Truck	55,000	5 years	165,000
Pumps	48,000	5 years	477,000
Pickup Truck	49,000	2 years	586,000
Manbus	85,000	5 years	255,000
Ambulance	106,000	10 years	106,000
Fire Truck	276,000	10 years	276,000
Compactor	301,000	10 years	301,000
Lowboy	106,000	LOM	106,000
Total			**99,571,000**
Lease Cost	**$/t mill feed**	-	**$0.85**

24.2.4 Processing

Process Plant

The process plant operating cost estimate is split into fixed and variable costs and is summarized in Table 24-17. The fixed costs presented here have been scaled from the base-case using the 6/10th rule. Total operating costs for the 40,000 t/d scenario are estimated to be $5.00/t of material processed.

Table 24-17: Process Plant Operating Cost Summary

Operating Costs	Costs per Year ($)	Costs per Tonne ($)
Fixed	16,625,100	1.15
Variable	55,391,500	3.85
Total	**72,016,600**	**5.00**

Fixed operating costs are summarized in Table 24-18. A combined labour cost of $0.63/t represents the largest component of the fixed costs, and 13% of the overall operating costs.

Table 24-18: Fixed Operating Cost Summary

Operating Costs	Costs per Year ($)	Costs per Tonne ($)
Labour		
Plant Management	867,400	0.06
Plant Operation	5,048,500	0.35
Plant Maintenance	1,932,300	0.13
Assay Lab	1,308,100	0.09
Safety Equipment	148,500	0.01
Plant Maintenance Spares	5,160,300	0.36
Electrical/Inst. Maintenance	2,160,000	0.15
Sub-Total	**16,625,100**	**1.15**

A breakdown of variable operating costs is given in Table 24-19. The largest components of the variable costs are reagents at $1.70/t, and steel (balls and liners) at $1.43/t. Together, these two areas account for 62% of the overall plant operating cost.

Table 24-19: Variable Operating Cost Summary

Variable Costs	Costs per Year ($)	Costs per Tonne ($)
Power	8,680,900	0.60
Reagents	24,449,900	1.70
Mill Balls	14,464,800	1.00
Liners (Crusher + Mills)	6,254,700	0.43
Piping	1,152,000	0.08
Assay Laboratory	129,900	0.01
Lubricants	259,200	0.02
Sub-Total	**55,391,500**	**3.85**

Labour

Labour costs have been calculated using typical plant staffing levels. Pay scales have been provided by analysis of neighbouring mines collective agreements. The plant schedule is assumed to be 12 hour shifts, with two shifts at the site and two shifts on leave at all times. Estimated labour breakdowns are given in Table 24-20 to Table 24-23.

Table 24-20: Plant Management Labour Cost Summary

Plant Management/Admin	No.	Shifts	Total	Basic Sal.	Benefits	Total/Person	Total
Plant Manager/Supt.	1	1	1	105,092	27,491	132,583	132,583
Met Clerk/Planner	1	2	2	50,086	14,212	64,298	128,596
Plant General Foreman	1	1	1	95,030	25,062	120,092	120,092
Metallurgical Engineer	1	1	1	95,030	25,062	120,092	120,092
Plant Metallurgist	1	4	4	71,999	19,502	91,501	366,005
Sub-total	-	-	**9**	-	-	-	**867,366**

Table 24-21: Plant Operations Labour Cost Summary

Plant Operation	No.	Shifts	Total	Basic Sal.	Benefits	Total/Person	Total
Plant Foreman (Shift)	1	4	4	84,968	22,632	107,600	430,400
Control Room Operators	2	4	8	84,968	22,632	107,600	860,800
Plant Operators	4	4	16	67,080	18,314	85,394	1,366,302
Reagent Operators	1	4	4	67,080	18,314	85,394	341,575
Labourers	6	4	24	67,080	18,314	85,394	2,049,453
Sub-total	-	-	**56**	-	-	-	**5,048,531**

Table 24-22: Plant Maintenance Labour Cost Summary

Maintenance	No.	Shifts	Total	Basic Sal.	Benefits	Total/Person	Total
Maintenance Lead	1	4	4	83,850	22,362	106,212	424,849
Millwright	1.5	4	6	78,260	21,013	99,273	595,636
Electrician	1.5	4	6	83,850	22,362	106,212	637,273
Instrument/Control Tech	1.5	2	3	71,999	19,502	91,501	274,504
Sub-total	-	-	**19**	-	-	-	**1,932,261**

Table 24-23: Assay Lab Labour Cost Summary

Laboratory	No	Shifts	Total	Basic Sal.	Benefits	Total/Person	Total
Chief Chemist	1	1	1	80,049	21,445	101,493	101,493
Chemist	1	2	2	67,080	18,314	85,394	170,788
Analytical	2	4	8	67,080	18,314	85,394	683,151
Samplers	1	4	4	69,316	18,854	88,170	352,679
Sub-total	-	-	**15**	-	-	-	**1,308,111**

In all, the mill is estimated to require total complement of 99 persons and an annual cost of $9.1 million or $0.63/t.

Reagents

Reagent costs are estimated using unit costs provided by vendors and consumption rates from the lab testwork and a summary is presented in Table 24-24. The total reagent cost amounts to $1.70/t of mill feed, or approximately 34% of the total operating cost for the plant. It should be noted that sodium cyanide alone represents roughly half of the cost of reagents.

Grinding Media

The cost of grinding balls and mill liners are estimated using quoted supply rates and consumption estimates from DRA's industry database (see Table 24-25).

Power

Electricity supply represents a significant operating cost, accounting for approximately 12% of the overall total. The cost of power is estimated on the basis of a connected rating of 37.4 MW and an 81% average load factor.

A power supply rate of $40/MWh has been provided by BC Hydro. This equates to an annual operating cost of $8.7 million or $0.60/t of mill feed.

Table 24-24: Summary Reagent Consumption Rates

Reagent	Consumption (g/t)	Consumption (t/a)	Unit Cost ($/t)	Annual Cost ($)	Annual Cost ($/t)
PAX	120	1,728	3,719	6,426,432	0.45
MIBC	45	648	4,057	2,628,936	0.18
Lime	400	5,760	225	1,296,000	0.09
NaCN	400	5,760	2,130	12,268,800	0.85
Activated Carbon	8	115	2,510	289,152	0.02
NaOH	5	72	1,430	102,960	0.01
Flocculant	20	288	3,500	1,008,000	0.07
$Na_2S_2O_3$	10	144	1,520	218,880	0.02
$CuSO_4$	2	29	2,442	70,330	0.00
HCl	15	216	650	140,400	0.01
Sub-total	-	-	-	**24,449,890**	**1.70**

Table 24-25: Grinding Media Consumption Rates and Operating Costs

	Annual Tonnage (t)	Annual Cost ($)
Mill Balls		
SAG (125 mm)	3,600	4,410,000
Ball (50 mm)	7,920	9,702,000
Regrind Mill	288	352,800
Total Cost	**11,520**	**14,464,800**
Liners		
Primary Crusher	216	1,460,446
SAG Mill	936	2,112,677
Ball Mill	1,296	1,856,505
Regrind Mill	576	825,113
Total Cost	**1,152**	**6,254,741**

Laboratory

Laboratory operating costs have been estimated using actual plant operations experience on similar plant. Lab operating costs are estimated at $130,000.

Maintenance Spares

The maintenance budget is calculated as a percentage of mechanical supply capital cost. Factors for each plant area were used to arrive at an overall budget for mechanical maintenance (replacement) spares of $5.16 million per year. An additional $2.16 million is allocated for electrical and instrumentation maintenance. Labour costs for replacement, installation, and maintenance are deemed to be covered in the labour allowance shown in earlier in this section.

Tailings Management Facility

Operating costs have been prepared for the following components of the TMF:

- TMF service road maintenance
- embankment maintenance
- manpower
- power for reclaim pumping – with a price of $40/MWh
- reclaim barge relocation
- TMF environmental compliance
- TMF engineering support and reporting.

The average annual TMF operating cost is $1.34 million. This equates to an operating cost for the TMF of $0.12/t of mill feed.

The cost estimate was compiled using information from similar projects, engineering experience, and equipment specifications.

24.2.5 General and Administrative

G&A costs include the cost of 13 salaried staff and 31 hourly personnel. Employees will be located in the immediate area and no camp is planned or required.

The G&A cost in the cash flow starts in Year -2 and increases in value until Year 2 when it reaches a maximum of $5.5 million per year. The labour cost is the single largest expense followed by communications and environmental monitoring. A staff burden rate of 23% was used with a 25% rate for hourly employees.

Table 24-26 shows the annual costs by category for Year 2, a representative average year.

Taxes and insurance are assumed paid by Head Office and not the mine site. The cost for G&A in Year 2 is $0.38/t of mill feed, but for the life-of-mine averages $0.43/t of mill feed. The higher unit cost is due to G&A costs being applied in the pre-strip period with no plant tonnage to offset, as the plant is not functional at that time.

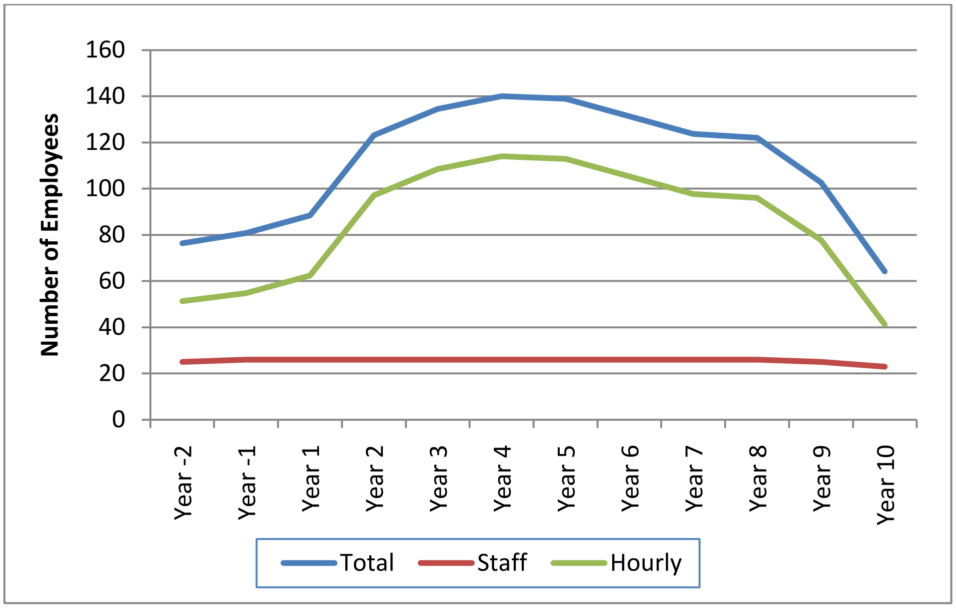
Table 24-26: G&A Cost Calculation (Year 2)

Category	Annual Costs (Year 2) ($)
Salaried Staff	1,218,000
Hourly Personnel	2,246,000
Site Operation and Maintenance Supplies	150,000
Site Power	100,000
Williams Lake Office	100,000
Information Systems (Hardware/Software)	100,000
Communications	300,000
Public/Community Relations	100,000
Recruitment and Training	200,000
Safety and Medical Supplies	70,000
Consultants	230,000
Legal and Audit Fees	150,000
Logistics	180,000
Office Supplies	75,000
Environmental Monitoring	250,000
Total G&A	5,469,000
Tonnage Milled (Year 2)	**14,400,000**
G&A Costs ($/t)	**0.38**

24.2.6 Mine Manpower

Manpower requirements for the Spanish Mountain Project vary from year to year by department. The detail on the manpower have been discussed in each section and the reader is referred to the appropriate section. Figure 24-2 shows the annual expected manpower levels.

Figure 24-2: Annual Manpower Levels



25 ECONOMIC ANALYSIS

25.1 Discounted Cash Flow Analysis

This assessment is preliminary in nature as it includes Inferred material which cannot be categorized as reserves at this time, and as such there is no certainty that the preliminary assessment and economics will be realized.

The tonnes and grades reported in Section 19 for the pit phases were used in the discounted cash flow (DCF) analysis for the Spanish Mountain Project. Table 25-1 outlines these again.

Table 25-1: Phase Tonnages and Grades

Phase	Plant Feed (t)	Diluted Au (g/t)	Total Waste (t)	Strip Ratio
Phase 1	16,989,200	0.72	15,779,000	0.9
Phase 2	25,388,000	0.46	50,659,500	2.0
Phase 3	24,315,800	0.57	37,859,600	1.6
Phase 4	50,213,000	0.43	125,828,000	2.5
Total	**116,906,000**	**0.51**	**230,126,100**	**2.0**

The completion of the trade-off study indicated that with higher gold prices, greater value could be obtained from a production rate of 40,000 t/d of plant feed. This was advanced in this study and is the chosen case for the project with the Financial Base Case gold price.

All prices quoted are in Canadian dollars unless otherwise noted. An exchange rate of C$1.10 to US$1.00 was used.

All pit design work was completed using what has been termed Engineering Case metal prices. The DCF analysis was completed using a higher gold price, which for this report was termed the financial base case metal price. Spanish Mountain Gold believes that the financial base case gold price reflects the longer-term metal price sentiment and gives an indication of the project potential. For comparison, the current three-year rolling average price has also been shown. Continued strength in the gold price has lead to an increasing rolling average price. The Canadian to American dollar exchange rate has also been charted in Table 25-2.

Table 25-2: Exchange Rate and Metal Prices

Metal/Rate	Unit	Engineering Base Case	Financial Base Case	Three Year Average Dec. 2007 – Dec.2010
Exchange	C$:US$	1.10	**1.10**	1.08
Gold	US$/oz	950	**1,100**	1,012

In the development of the operating costs for the DCF, the impact of leasing was considered. The potential economics improved as a result of its inclusion and were adopted as the chosen case. The results of the DCF for the 40,000 t/d case with and without leasing have been shown in Table 25-3.

Table 25-3: Discounted Cash Flow Results

Cost Category	Units	Without Leasing 40,000 (t/d)	With Leasing 40,000 (t/d)
Operating Costs			
Open Pit Mining	($ M)	438	438
Lease Costs	($ M)	-	99
Processing	($ M)	598	598
G&A	($ M)	50	50
Sub-total Operating Costs	($ M)	1,086	1,185
Capital Costs			
Open Pit Mining	($ M)	86	-
Processing	($ M)	215	215
Infrastructure–Site	($ M)	51	51
Infrastructure–Tailings	($ M)	37	37
Environmental Costs	($ M)	18	18
Indirect	($ M)	79	70
Contingency	($ M)	85	72
Sub-total Capital Costs	($ M)	571	463
Revenue (after refining, payables)	($ M)	2,060	2,060
Net Present Value (NPV)			
NPV @ 0%	($ M)	404	411
NPV @ 5%	**($ M)**	**193**	**209**
NPV @ 8%	($ M)	106	125
IRR	**(%)**	**13.2**	**14.7**
Payback Period	Years (Year paid)	4.3 (Year 5)	4.1 (Year 5)

The financial base case gold price is US$1,100/oz. Payables for gold were increased to 99.5% from the 98.5% used in the trade off study. A royalty of 1% was not applied for the calculation of NPV in the cash flow.

The results of the DCF for the 40,000 t/d with leasing case indicated that the project has a pre-tax NPV of $209 million at a discount rate of 5% with an IRR of 14.7%. This is an improvement of $16 million in the pre-tax NPV over the no-leasing case, which had an IRR of only 13.2%. Payback on the project from the start of commercial production is 4.1 years.

Table 25-4: Metal Production Statistics, Cash Cost Calculations, and Key Economic Parameters

Item	Indicator	Units	Value
Gold	Average annual production	oz	172,400
	Initial 5-year average annual production	oz	213,800
	Total LOM production	Moz	1.72
Cash Costs	Average LOM gold cash cost	US$/oz	625
	Initial 5-year average gold cash cost	US$/oz	570
Key Parameters	Operating cost	$/t plant feed	10.14
	Mine life	years	10
	Average plant feed grade	g/t	0.51
	Overall gold recovery	%	90
	Initial capital costs	$ M	447
	Total capital costs	$ M	463

25.2 Sensitivity Analysis

The project sensitivity to various inputs was examined on the 40,000 t/d case with leasing. The items that varied were:

- gold price
- gold recovery
- capital costs
- operating cost.

The results of that analysis have been shown in two spider diagrams. Note that the recovery was held to 100%, which accounts for the irregular line in the graph for recovery in the +20% bin. The spider graphs are shown in Figure 25-1 and Figure 25-2.

Figure 25-1: Spider Graph of NPV at 5%

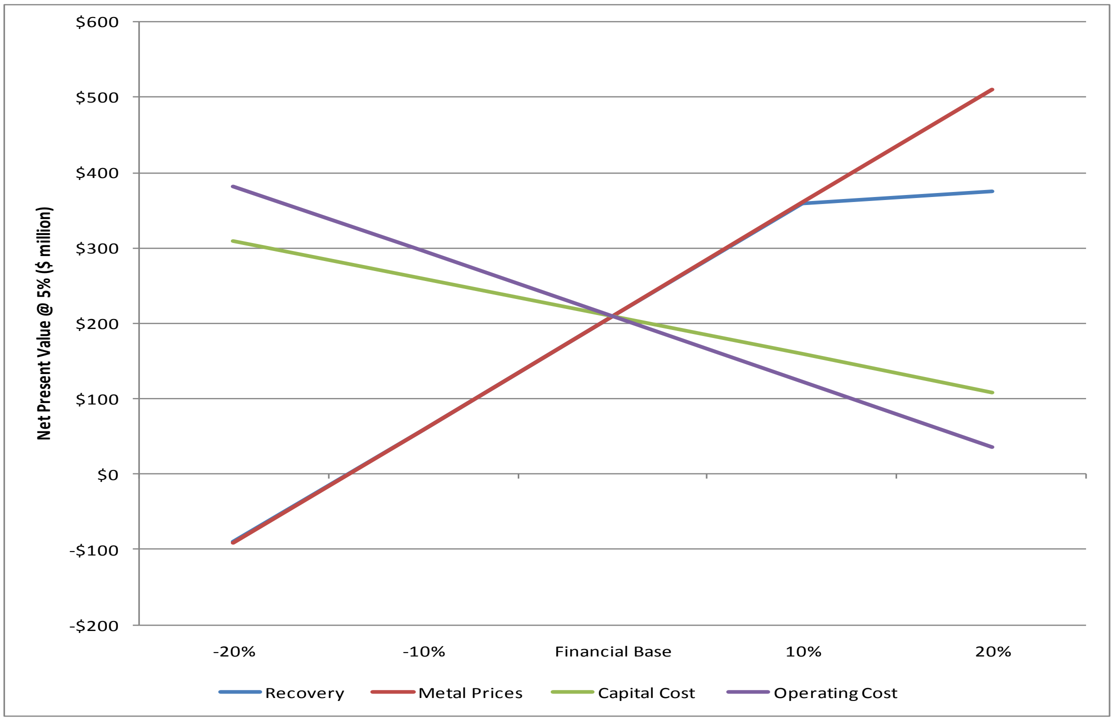
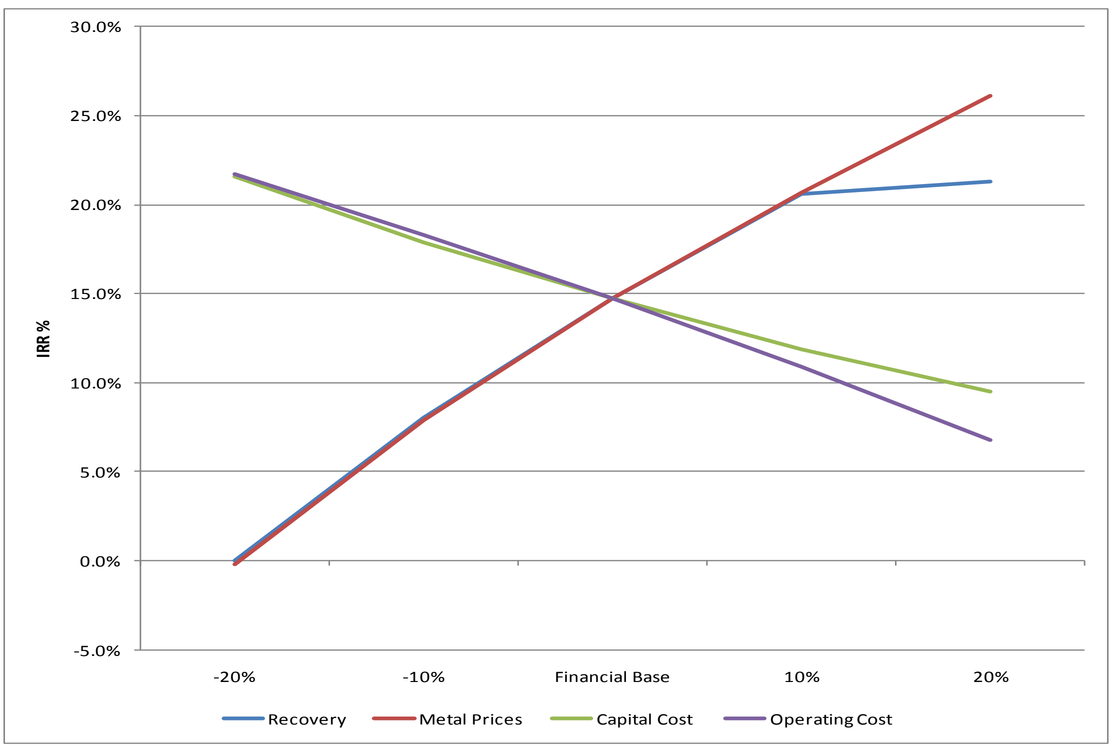


Figure 25-2: Spider Graph of IRR



What this indicated was that the project is most sensitive to gold recovery and metal prices, and least sensitive to capital costs. With the mining equipment fully leased, there is a further reduction in the impact of capital on the overall project.

Mining costs currently consider haulage of material to the TMF for storage. Changes in the percentages of PAG material may alter the haulage profiles, but given the wrap around nature of the waste dumps, a significant increase in cost is not expected unless great quantities required storage in the TMF.

Metallurgical recovery, as described in Section 16 of this report, is expected to be stable over a wide range of feed grades. Therefore, while a sensitivity exists, actual practice may show less fluctuation than considered in this analysis. A 20% change in recovery would equate to approximately 72% recovery from the existing testwork value of 90%. While further study is required, that range of fluctuation is not expected.

Table 25-5: Sensitivity Analysis – NPV at 5% Discount Rate

Sensitivity	Unit	Recovery	Metal Prices	Capital Costs	Operating Costs
-20%	($M)	-90	-92	309	382
-10%	($M)	59	58	259	296
Financial Base	**($M)**	**209**	**209**	**209**	**209**
+10%	($M)	359	360	159	123
+20%	($M)	375	511	109	36

Table 25-6: Sensitivity Analysis – IRR

Sensitivity	Unit	Recovery	Metal Prices	Capital Costs	Operating Costs
-20%	(%)	0.0	-0.2	21.6	21.7
-10%	(%)	8.0	7.9	17.9	18.3
Financial Base	**(%)**	**14.7**	**14.7**	**14.7**	**14.7**
+10%	(%)	20.6	20.7	11.9	10.9
+20%	(%)	21.3	26.1	9.5	6.8

This leaves the greatest sensitivity in the project to gold price. With the current Financial Base price of US$1,100/oz, this is still $303/oz less than the spot price of US$1,403 as of 13 December 2010. Market conditions will determine this value, with the management of the project to focus on those items that can be controlled on site; recovery, capital costs, and operating costs.

The sensitivity of the current project to metal prices was also examined. The pit design and schedule did not change, only the value of the gold. The results have been shown in Table 25-7.

Table 25-7: Sensitivity Analysis – Gold Price Impact

Gold Price (US$/oz)	NPV ($M) @ 0%	NPV ($M) @ 5%	NPV ($M) @ 8%	IRR%
$950	128	4	-47	5.2
$1,000	222	72	10	8.6
$1,050	317	141	68	11.7
$1,100 (Financial Base)	**411**	**209**	**125**	**14.7**
$1,150	505	278	182	17.5
$1,200	600	346	240	20.2
$1,250	694	415	297	22.7
$1,300	788	483	355	25.2

25.3 Engineering Base Case Price Sensitivity

The open pit design was based on the use of a gold price of US$950/oz to allow for expected price fluctuations. Market conditions have to date indicated a positive increase in the price of gold since initiation of the study, with longer-term forecasts remaining strong. To determine the impact of a higher price on open pit potential, a series of pit shells were developed for gold prices in excess of US$950/oz. These provided Spanish Mountain Gold with guidance on the potential expansion capability and drill targets for potential resource improvement.

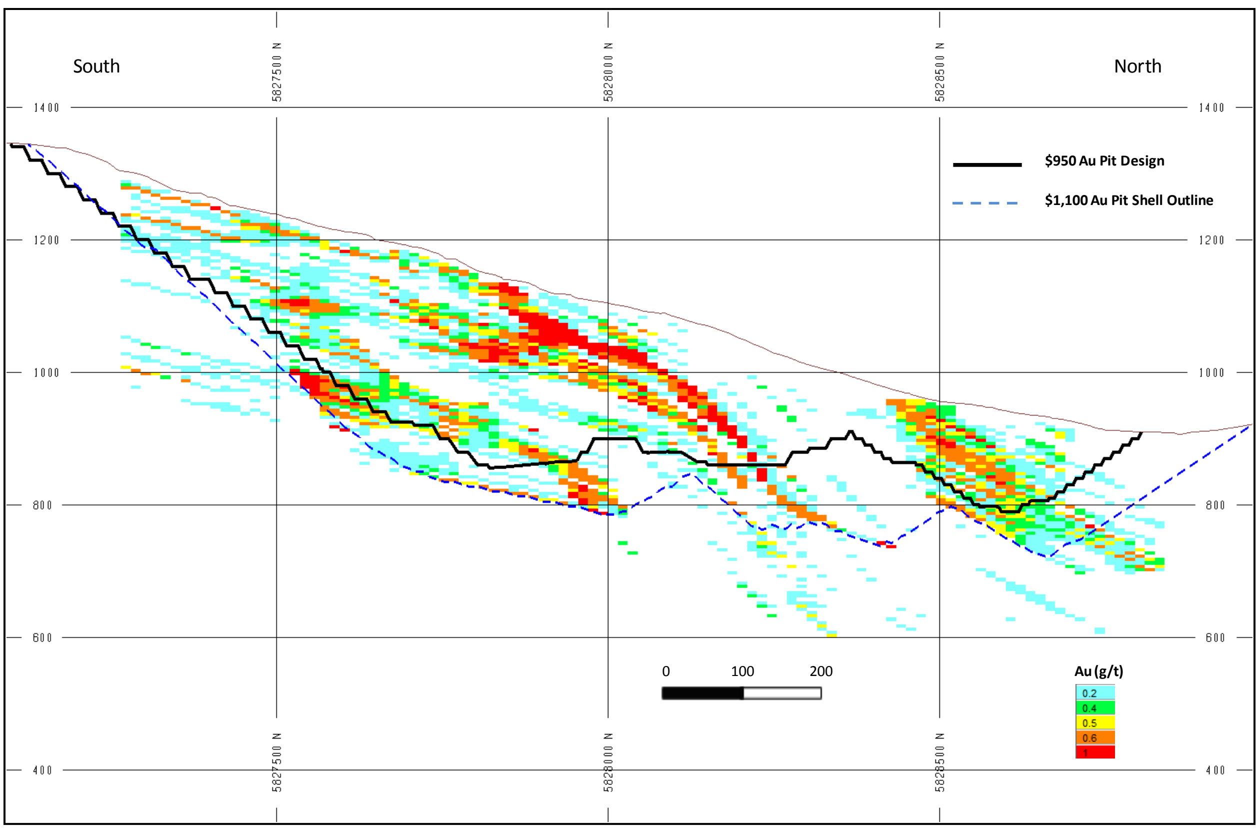
Using the initial parameters from the trade-off study for the 40,000 t/d case, four additional pits were developed. These were compared to the original US$950 pit shell in Table 25-8.

Table 25-8: Sensitivity Analysis – Pit Shell Size to Gold Price

Item	Units	US$950/oz	US$975/oz	US$1,000/oz	US$1,050/oz	US$1,100/oz
Mining cutoff	g/t	0.24	0.23	0.23	0.22	0.21
Milling cutoff	g/t	0.19	0.18	0.18	0.17	0.16
Plant feed	Mt	158.6	170.5	175.8	189.4	203.8
Plant feed	g/t	0.49	0.48	0.48	0.46	0.44
Waste	Mt	278.3	301.9	304.5	310.2	321.3
Total material	Mt	436.9	472.4	480.3	499.6	525.1
Strip ratio		1.76	1.77	1.73	1.64	1.58
In situ gold	Moz	2.52	2.65	2.69	2.80	2.91

What this analysis indicated was that with a gold price of US$1,100/oz, the resulting shell provided 29% more material suitable for plant feed with a grade of 0.44 g/t. Contained ounces of gold within this shell increased from 2.5 Moz to 2.9 Moz. It should be noted that the pit shell ounces are not the exact ounces that would be extracted in the final pit design. A reduction would be expected due to pit bottom widths, ramps encroaching on the pit bottom, and other practical design considerations. The Lerch-Grossman routine creates a shell with a point that is not practical to mine in almost all the cases. This is also noted by the reduction in the ounces in the US$950 shell. Expected recovered ounces for the shell were 2.27 million, but the final design resulted in only 1.72 Moz. It does, however, provide a good indication of the potential that may exist with additional drilling and detailed design work. The US$1,100/oz pit shell has been shown in cross-section with the existing pit design in Figure 25-3. The pit shell is indicated by the dashed line.

Figure 25-3: Pit Design vs. US$1,100/oz Pit Shell



26 PROJECT IMPLEMENTATION PLAN

The Project Implementation Plan (PIP) for the Spanish Mountain Project is shown in Figure 26-1.

This schedule considers a logical path forward on the development path for the Spanish Mountain Project. The four main areas considered in this schedule include:

1. Exploration
2. Environmental and Socioeconomic Studies and Permitting
3. Engineering Studies and Mine Development
4. Plant Commissioning.

26.1.1 Exploration

Exploration will entail a drill program in the first quarter of 2011 to expand on the existing resource base as well as upgrade the classification of the Inferred material to Indicated or Measured. This upgrading of resources will allow their inclusion in engineering studies of greater detail including Prefeasibility and Feasibility. Samples from this program will be used in further optimization of the metallurgical testwork. The updated drilling will be incorporated into a new geologic model for use in the Prefeasibility Study.

Upon completion of the prefeasibility study, further drilling may be required for resource definition, metallurgical testwork, geotechnical analysis, and condemnation drilling. The completion of this work will result in the development of the feasibility resource model for use in the feasibility study and initial mine production.

26.1.2 Environmental and Socioeconomic Studies and Permitting

Spanish Mountain Gold considers that consultation with First Nation Groups must continue to ensure that they are an active and informed participant in the development of the Spanish Mountain Project. This is expected through the entire study period and into production.

With the completion of the PEA study, a Project Description will be submitted to the BC Environmental Assessment Office. At the same time, the detailed environmental and socioeconomic baseline studies will be initiated. Two years are required for this work to be complete. This is planned to commence the first quarter of 2011.

Figure 26-1: Project Implementation Plan Schedule

		Spanish Mountain Implementation Plan Schedule																			
		2011				2012				2013				2014				2015			
Task Name	**Time (years)**	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4
Exploration		----	----	----	----	----	----														
Drill Program - Resource expansion/Classification Upgrade	0.50	■	■																		
Resource and Metallurgy Update	0.25			■																	
Feasibility Study Drilling (Resource, Metallurgical, Geotech)	0.25				■																
Feasibility Resource Update	0.25					■															
Environmental/Socioeconomic Studies/Permitting		----	----	----	----	----	----	----	----	----	----	----	----	----	----	----	----	----	----	----	----
First Nation Consultation		■	■	■	■	■	■	■	■	■	■	■	■	■	■	■	■	■	■	■	■
Project Description Submission	0.25	■																			
Detailed Environmental and Socio Economic Baseline Studies	2.00	■	■	■	■	■	■	■	■												
Metal Leaching/Acid Rock Drainage Characterization	0.75	■	■	■																	
Draft Application Information Requirements (AIR)	0.50		■	■																	
Comprehensive Study Report (CSR)	0.50		■	■																	
Regulatory Meetings/Public Open House	0.75			■	■	■															
Final AIR/CSR Issued	0.50						■	■													
Impact Assessment	0.50								■	■											
Provincial Environmental Assessment Certificate	0.25										■										
Federal Responsible Authorities - Notice of Decision	0.25										■										
Engineering Studies/Mine Development			----	----	----	----	----	----	----	----	----	----	----	----	----	----	----	----	----		
Metallurgical Optimization	0.50		■	■																	
Prefeasibility Study	0.50		■	■																	
Feasibility Study	0.50				■	■															
Project Approval Decision	0.25						■														
Financing	1.00						■	■	■	■											
Basic Engineering/Site Preparation/Purchase Long Lead Items	0.50						■	■	■	■											
EPCM	1.50									■	■	■	■	■	■						
Infrastructure Development	1.50										■	■	■	■	■	■					
Open Pit Prestrip and Plant Feed Stockpiling	2.00										■	■	■	■	■	■	■	■			
Plant Commissioning																		----	----		
Plant Startup	0.25																	■			
Ramp-up Operations	0.50																	■	■		

From the PEA study, a recommendation for further waste rock classification for acid rock drainage was included. In addition, metal leaching testwork is also recommended to ensure no aspects with the potential to be detrimental to the environment would be overlooked. This work would commence in the first quarter of 2011 and continue to the end of the third quarter of 2011. The results would form the basis for the Prefeasibility and Feasibility studies going forward.

An extensive period of concurrent applications to Government agencies will occur in 2011, 2012 and half of 2013. This will include submission of the baseline data and study results. In addition, regulatory meetings and public open houses will form part of the process to ensure that feedback is provided to the public and the public has an opportunity to respond constructively to the projects development.

Completion of the process is expected at the end of the second quarter of 2013 with the receipt of required permits and authorizations. A full two and half year process is expected prior to approval being provided.

26.1.3 Engineering Studies and Mine Development

The completion of this PEA study represents the initiation of a series of detailed studies required to develop sufficient technical information for construction of the Spanish Mountain Project. As mentioned in the exploration portion of the PIP, additional metallurgical optimization will be required with the newly drilled samples from the first quarter 2011 drill program.

That information will then be fed into a prefeasibility study utilizing Measured and Indicated material only. Further production rate studies will be completed as trade-offs in the Prefeasibility to ensure the recommended production rate represents the rate of highest economic benefit to the project. This will be accomplished using Measured and Indicated material only from the updated Geologic Model. Recommendations from this study will be used to guide the feasibility study.

The feasibility study will detail the recommended option from the prefeasibility study. The results of that work will form the basis for the future mine, included expected economics. On this study, financing for the project can be procured to advance it towards construction if recommended.

Pending approval of all necessary permits and favourable economics from the feasibility study, Spanish Mountain Gold Board of Directors would be expected to make a positive decision to advance the project to construction.

Basic engineering of the plant, tailings, and infrastructure would then commence in earnest. This also includes the purchase of long lead-time items such as SAG mills. Currently it is envisaged that this would start in the second quarter of 2012

Final engineering and procurement of material would start the first quarter of 2013. The construction of the plant and infrastructure would be expected to start the second quarter of 2013.

Prestripping of the open pit and building of a plant stockpile is expected to commence the second quarter of 2013. The current mine plan envisages a two-year prestripping period which includes the mine assisting the construction of the tailings embankments.

Plant construction is expected to last 18 months.

26.1.4 Plant Commissioning

Plant commissioning would be the first quarter of 2016 with a six-month ramp up period. At the end of the ramp up period, the plant would be achieving the required 40,000 t/d or production.

26.2 Infrastructure and Site Layout

The infrastructure and the site plan design are based on information from published data and from previous work on similar operations. Several assumptions have been made in the costing of this portion of the Project and subsequent testing will provide more accurate data for refining the site plan design and associated infrastructure costs.

The site for the mill and operations was chosen for the following reasons:

- relative proximity to the open pit locations
- site is on top of a relatively flat height of land
- proximity to existing electrical infrastructure and water sources.

For the purpose of this study, water sources required for operations will be supplied by nearby Spanish Lake, recycle water from the tailings area, and water from mine dewatering operations.

This study assumed that the site 15 kV aerial circuit would supply the electrical power source for infrastructure in the immediate vicinity of the site. Infrastructure remote to the site will be supplied by the local BC Hydro distribution system.

27 INTERPRETATION AND CONCLUSIONS

27.1 Geology

Exploration at the Spanish Mountain property has led to the discovery of disseminated gold within sedimentary rocks, and to a lesser extent, free gold in quartz veins.

Using diamond drill hole data from holes completed between 2005 and 2009, AGP modelled grade shells based on laterally continuous zones of drill hole data greater than 0.6 g/t Au and lithological units representing a simplified stratigraphy. AGP interpolated grade in blocks inside of the grade shells and within the lithological domains.

AGP estimated that the Spanish Mountain deposit contains 4.9 Mt averaging 1.04 g/t Au in the Measured category, 72.5 Mt averaging 0.5 g/t Au in the Indicated category, and 39.5 Mt averaging 0.47 g/t Au in the Inferred category.

27.2 Geotechnical

The preliminary engineering geology of the Spanish Mountain deposit has been summarized to provide a basis for scoping level mine planning and preliminary economic assessments. BGC has developed a basic description of the expected geologic materials of the resource area from available maps, geologic descriptions by Spanish Mountain Gold, core hole data, and field review.

The five preliminary geotechnical units for mine design are siltstone, argillite, greywacke, conglomerate, and fault zones. Relatively limited data is available regarding the rock mass strength and the geologic structure in the Spanish Mountain deposit. The main limitations to the data are as follows:

- All of the core available for inspection has already been cut in half. In order to conduct accurate geotechnical investigations the core should be intact.
- In general, the joint spacing and rock strength for all units are not well defined, as previous logging techniques were insensitive to these properties.
- Structural geologic information is relatively sparse.

Sufficient data has been compiled regarding geotechnical strengths of the primary rock types to provide a range of potential pit wall angles for use in the preliminary economic assessment. However, in order to develop the slope design angles presented in this report, numerous assumptions had to be made about the potential primary controls on slope stability, the geology, the strength of the rock mass, the groundwater pressures and the potential failure mechanism. The following assumptions were made:

- Inter-ramp slope angles could be limited due to structurally controlled failures along continuous bedding.
- Anisotropy of the rock mass was not considered in the generic (i.e., rock mass) stability analyses conducted.
- Groundwater pressures were assumed to be a function of the lithostatic stress.

The results of the kinematic and rock mass stability assessments are described in the following sections.

27.2.1 Structurally Controlled Instability

Based on the structural data reviewed, the following observations have been made:

- Bedding orientations exhibit a substantial (20°-50°) rotation between the Main and North Zones, suggesting a structural domain boundary may be present between the two zones.
- Bedding in the Main Zone dips primarily towards the southwest (194°) at an average angle of about 30°. A second weaker concentration of bedding planes is oriented almost due north at an angle of about 40°. The bedding discontinuities could be a primary structural control on inter-ramp slopes in the north wall of the proposed Main Zone pit, if the bedding is continuous.
- Bedding in the North Zone is highly variable, with a wide range of dip directions from the northeast to the southwest. Three main sets have been identified with average dip angles ranging from 32° to 43°.
- Based on information obtained from the structural geologic mapping and oriented core, south-southwest and north-northeast facing pit walls could be subject to structurally controlled instability due to the strong presence of bedding.
- In general, the core orientation data appears to be supported by the surface mapping data although there is evidence of significant folding and variation of bedding with depth. This is supported by our observations from shallow test pits in rock in the project area.

Structural discontinuities at the inter-ramp scale could control achievable inter-ramp angles on north facing slopes in the Main Zone, and for northeast to southwest facing slopes in the North Zone. At this preliminary stage of design, it is recommended that bedding should not be undercut where the average dip is greater than 30°, in order to minimize the potential for structurally controlled instability. This applies to Design Sector MZ-180 (slope azimuth 150° to 210°) in the Main Zone and Design Sectors NZ-238, NZ-315 and NZ-025 (slope azimuth 190° to 065°) in the North Zone. Lower hemisphere equal area stereonets showing the structural discontinuity populations and the preliminary design sectors are included in Appendix C.

Despite considerable scatter in the bedding orientations, both the oriented core and surface mapping data in the Main Zone and North Zones identify two to three prominent bedding orientations. Depending on the local continuity of the bedding and the shear strength of these discontinuities, bench scale failures may occur for these pit wall orientations, and occasional wide berms may be required to contain the failures. However, until additional information on the continuity and spacing of the bedding is available the confidence level of the bench designs (i.e., their ability to contain bench-scale instability) will be low. The variability of the bedding orientations needs to be further evaluated to determine whether this has an overall positive or negative effect on the achievable inter-ramp and overall slope angles.

27.2.2 Rock Mass Instability

Based on the estimated rock mass strength of the argillites in the Main Zone, overall pit wall angles of 32° to 43° are predicted to be feasible for pit wall heights between 250 m and 500 m. Based on the estimated rock mass strength of the greywacke, significantly steeper overall slopes could be achieved; however, it appears that the critical south wall will be primarily in the footwall argillites. Note that the predicted achievable overall slope angles assume that there is no structural control on the potential failures. Unfavourably oriented geologic structures are likely present locally, and additional structural geologic investigations/interpretations are necessary to gain greater confidence in these recommended wall angles.

Based on the estimated rock mass strength of the siltstone in the North Zone, and assuming "dry" conditions, overall pit wall angles ranging from 42° to 55° could potentially be achieved for slope heights from 100 m to 200 m. However, partially saturated conditions are likely more reasonably assumed due to the presence of Spanish Creek nearby, in which case considerably shallower pit wall angles of between 37° and 48° are predicted. Regardless of the assumptions with respect to groundwater pressures in the North Zone, a high degree of depressurization will be required to achieve reasonable slope angles in the siltstone. Groundwater pressures will need to be more accurately quantified in the proposed pit walls before greater confidence can be gained in the design angles for these materials.

27.2.3 Data Reliability

Engineering geology interpretations and pit wall design angles presented in this report are based on adequate information for scoping level designs, but should be considered preliminary. Where appropriate, geological features identified should be verified and validated with additional fieldwork and interpretation. Data used to provide initial quantitative estimates of the rock mass properties have primarily been collected within the Main and North zones, and exploration drilling has concentrated on the mineralized horizon. This data may not accurately reflect the rock mass comprising the final open pit walls. The geomechanical properties of the rocks outside the ore zone could control the final excavation geometry and have a significant impact on mining economics. Recommendations for improving the quality of available data and reliability of the engineering geology interpretations for more detailed mine designs are provided in Section 28.

27.3 Mining

It should be noted that the project contains Inferred Mineral Resources that are too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. The operational plan described in this study is preliminary in nature and there is no certainty that the operational life of mine plans can be realized. The level of study undertaken is considered equivalent to a scoping study with estimation accuracy of around +30%/-25%. Considerable additional detailed planning and design is required to confirm the production schedules and cost estimates. However, AGP believes the study reflects a reasonable assessment of the open pit mining potential based on the current information and data available.

The economic analysis indicated that with a gold price of US$1,100/oz the project has a potential NPV of $209 million with a discount rate of 5%. The IRR for that case was 14.7% on a pre-tax basis. Potential payback of project capital is less than five years with payback occurring in the fifth year from start of milling operations.

The open pit mine will provide to the process plant a total of 116.9 Mt of feed grading 0.51 g/t. The final year of processing is estimated to be Year 10. Waste from the mine amounts to 230.1 Mt for an overall strip ratio of 1.97:1. Plant feed will be comprised of 4.9 Mt grading 1.04 g/t in the Measured category, 72.5 Mt in the indicated category grading 0.50 g/t. This totals 77.4 Mt grading 0.53 g/t. Inferred resource mined amounts to 39.5 Mt grading 0.46 g/t. The grades indicated are diluted grades and expected to comprise the plant feed.

Open pit mining commences in Year -2 with pre-stripping, stockpiling of plant feed material. NAG material is mined and used for construction of the TMF. Plant production commences in Year 1 and continues at 40,000 t/d until Year 7 where the tonnage tapers off until the mine is complete in Year 10. In Year 7, Phase 2 (North Zone Pit) is complete and will be used for storage of the remaining waste material. This assists the overall mine costs by reducing the haulage requirements and also helps the end leave reclamation position by partially backfilling the open pit.

The waste is broken into AG, PAG, and NAG. Less than 1% of the material is AG, 72% is PAG and the remaining 28% is NAG. NAG material will be used for construction of the TMF embankments in addition to comingling with the PAG material. A portion of the PAG material will be stored in the TMF. Approximately 6.8% of the PAG total will be sent the TMF for subaqueous storage. Further work is required to better define the AG/PAG/NAG split as the model used in the PEA is based on limited information.

Production in the pit will be completed with two diesel rotary drills (200 mm), two electric hydraulic shovels (21 m^3), and a large front-end loader (21 m^3) as backup. A fleet of thirteen 180-tonne haulage trucks will haul waste material. The life-of-mine operating cost is forecast to be $1.26/t of material moved.

Additional geotechnical work is required to assist in increasing the slope angles if possible. This would have the benefit of reducing waste or allowing the pit to dive deeper after material below the existing pit. Geotechnical slope stabilities need to be examined for the west and east dumps which are on the edge of the open pit. With the low slope angle, it was assumed that this would not be an issue but it needs to be verified.

Condemnation drilling needs to be completed in all waste dump locations in addition to the plant site.

27.4 Metallurgy

Samples from the Spanish Mountain deposit have been submitted for grindability testing by standard industry methods including Bond Work Index and JK Drop Weight. Results indicate that the ore is of medium hardness and is amenable to conventional processing in a SAG mill followed by a closed circuit ball mill arrangement.

Metallurgical testwork on three composite samples from hole #865 of the deposit have indicated that the gold in the ore upgrades well by gravity or conventional sulphide froth flotation at a moderately coarse primary grind P_{80} of 184 µm. Regrinding of the concentrate to a P_{80} of 20 µm has been shown to be successful at rendering the gold in the concentrate amenable to recovery by CIL cyanidation.

A process plant design has been developed based on the available testwork data that combines industry standard unit process operations into a flowsheet that can be considered to be of low- to medium-complexity.

Capital and operating costs for the proposed plant have been developed first at the base case plant throughput of 25,000 t/d and then scaled to the design rate of 40,000 t/d.

27.5 Waste and Water Management

The work completed to date suggests that the current waste and water management concept is practicable and should be carried forward to the next level of design.

The conceptual design presented here has the potential for optimization and the geotechnical conditions, design basis, and operating criteria are further refined. Additional study will be required in the next phase of design to ensure that the TMF option incorporated in future studies is the most appropriate option based on cost, geotechnical conditions, and environmental impacts. This will involve further identifications and evaluations of the risks and opportunities of this concept and any additional concept being considered.

28 RECOMMENDATIONS

28.1 Geology

AGP recommends additional diamond drilling of 20 to 30 holes (10,000 to 15,000 m), focused in the north zone and in the main zone at depth. Additional drill hole data may allow Inferred material to be up-classed to Indicated assuming that increased drilling will result in an increase in understanding on the geologic controls on mineralization. Anticipated costs for this program could range from $2.5 to $3 million.

An effort to refine the current 3D model by incorporating structural information such as faults is recommended.

AGP recommends the collection of samples from new drill holes at regular intervals for specific gravity determination. Testing should consider moisture content and results should be expressed as dry bulk specific gravity.

AGP also recommends a selection of 100 pulps from NQ samples that were analyzed by Ecotech be sent to ALS for analysis to test for bias in grade.

28.2 Geotechnical

28.2.1 General

The interpretations of this report are preliminary and require additional validation and testing with higher quality data before they can be applied to higher-level design studies. For prefeasibility design studies, greater confidence in the geotechnical input parameters will be required and the preliminary geotechnical model will need to be updated with additional data. A series of recommended data collection and interpretation tasks are outlined in the following sections.

28.2.2 Outcrop Mapping

Mapping of additional exposed rock outcrops along drill roads or other access roads could provide important data on discontinuity orientation, character, and continuity; which are all critical for rock excavation design. Further information on the quality of the rock mass and the character and thickness of the overburden and/or oxidized rocks should also be collected during outcrop mapping.

28.2.3 Geotechnical Core Logging

Seven dedicated geotechnical core holes were drilled in 2010 targeting the proposed PEA level mining excavation. Core orientation techniques were employed to determine sub-surface geologic discontinuity orientations. These holes targeted waste rock outside of the ore zone to determine the geotechnical properties of the rock mass forming the pit walls. This information should be compared with surface mapping information and used to revise the structural domain boundaries, and to conduct kinematic stability analyses of the proposed open pit.

The existing core hole database does not provide sufficient geotechnical data to characterize the rock mass of the resource area according to standard rock mechanics techniques. The core logging procedures should be modified to provide complete parameters for RMR (Bieniawski, 1976). We recommend that the joint roughness characteristic (JRC), as defined by Barton and Choubey (1977) be collected on the discontinuities logged as part of the core orientation work. The JRC can be used to estimate the friction angle of the discontinuity, which will assist in kinematic stability analyses for slope designs.

28.2.4 Point Load Testing

Experiences at other mining properties and published literature have indicated that alteration may have a significant impact on the intact strength of the rocks of the resource area. The potential for further division of the geotechnical units according to alteration may be evaluated through a point load-testing program. The point load test is a simple and rapid method of determining an index value, which can be related to the intact strength of a rock sample. This index may be used to relatively compare intact strength variation according to alteration.

Spanish Mountain Gold should undertake a point load-testing program as part of its next exploration-drilling program. The added effort is minimal, requiring the rental or purchase of a point load test machine, the selection of samples according to lithology and alteration type, and completing the testing itself.

Discrepancies between estimated of UCS from field hardness grade, point load testing and laboratory testing should also be resolved in the next phase of work.

28.2.5 Laboratory Testing

Uniaxial compressive strength testing, direct shear testing of discontinuities, Brazilian tensile strength testing, and index testing of discontinuity infill should be conducted on select samples to provide a basis for geotechnical analysis and design parameters. These samples should be collected from dedicated geotechnical drill holes so that appropriate materials are sampled, and to avoid conflicts with exploration sampling and assaying.

28.2.6 Hydrogeologic Evaluations

Hydrogeological testing (packer testing) and instrumentation (i.e., piezometers) should be installed in select holes to provide basic data for groundwater modelling and excavation dewatering/depressurization simulations. This information will be useful in subsequent geotechnical evaluations, to determine the feasibility of dewatering the proposed pit.

28.2.7 Costs

Depending on the actual drilling time required to complete the geotechnical drilling, anticipated costs for pre-feasibility level investigations could range from about $500,000 to $600,000. Disbursements for the field and laboratory testing components of the work typically range from $75,000 to $125,000 and are included in this total.

28.3 Open Pit Mining

Detailed ARD information is required for the open pit. An initial model was developed with the information available but it is limited. Metal leaching potential in addition to acid generating or buffering capability needs to be modelled. With this information, proper scheduling of waste material for TMF construction and waste dump design can be developed. As haulage represents the largest portion of the mine operating cost, proper design of the waste haulage impacts both on operating cost as well as capital requirements.

Additional geotechnical information and slope recommendations are required for the open pit design. Special emphasis should be placed on the area adjacent to Spanish Creek as well as the south slope. The area near the creek may require additional dewatering or flatter slopes due to the presence of the creek. The south slope with its flat angle represents the greatest area for increased waste and potential slope failures. This needs to be detailed.

Waste dump slopes need be confirmed for base stability. The west and east dump lie adjacent to the pit and represent short haul opportunities. Their stability needs to be confirmed.

Further upgrading of material within the known boundaries should be completed. Drill proving of this material has the potential to expand the pit size and provide additional plant feed.

Blasting analysis work should commence. The proper design of blasting and material size will assist in the appropriate sizing of the primary crusher and help in the determination of the mining cost.

Condemnation drilling needs to be completed in all waste dump areas to verify that no additional material exists, particularly beneath the proposed plant location and the side waste dumps.

Additional drilling on the south end of the deposit should be completed also to determine if more plant feed material exists. If this were the case, the potential to push the south wall further south and daylight would help in the concern on the south slope by reducing its height.

28.4 Metallurgy

Additional metallurgical testing is recommended in the following areas:

- *Variability Testwork* – Additional metallurgical samples should be tested representing the various zones/lithologies in the deposit. This is particularly important for the gravity/flotation/CIL section of the plant as all of the work presented here comes from a single drill hole.
- *Gravity Testwork* – At present, gravity recovery is included in the flowsheet, but additional work is required to determine the ideal location for gravity in the flowsheet, as well as the opportunity for processing the gravity concentrate separately from the flotation concentrate. It should also be investigated if the gravity circuit can be eliminated altogether to reduce capital costs.
- *Grind Size Optimization* – Additional testwork is recommended to further characterize the relationship between primary grind size and flotation recovery. An opportunity may exist to coarsen the grind and reduce capital and operating costs without significantly affecting overall gold recovery.
- *Cyanide Consumption* – Roughly half of the cost of reagents, and 17% of the overall operating cost is the result of sodium cyanide. Additional work should focus on accurately forecasting what that consumption (and variability in consumption) is likely to be, and on methods for reducing its use.
- *Pre-concentration* – The opportunity exists to reject waste material prior to the grinding circuit and thereby reduce both capital and operating costs. Possible methods include optical sorting, jigging, and dense media separation. Mineralogical and metallurgical testwork should be carried out to determine if the Spanish Mountain ore is amenable to upgrading by one of these methods.

- *Gravity Recovery on 1st Cleaner Tailings* – Approximately 1.5% of the contained gold in the ore is lost to the 1st cleaner tailings stream. The potential for gravity or flotation recovery of part of the value from this stream should be investigated.
- *Tailings treatment/characterization* – Additional work is required to verify the operating parameters for the cyanide destruction circuit as well as to characterize the tailings as non-acid generating and stable for conventional disposal.

28.5 Infrastructure and Site Layout

Additional testing and data is required to further define the infrastructure and site layout requirements and associated costs in areas of:

- Topographical information to provide more detail for infrastructure design including power line routing, access road rehabilitation and building placement.
- Geotechnical testing and data to further develop construction requirements for both the mill/services site.
- Hydrogeological testing to determine the effect on infrastructure requirements.
- Bathymetry information and water quality testing on water sources surrounding the mine site (lakes and rivers) to determine available volumes and quality of water required to support the mine mill and services infrastructure.
- Further electrical study is necessary in order to validate the conceptual design. BC Hydro can perform a system impact analysis in order to model the affect of this connection and comment on any stability concerns.

28.6 Waste and Water Management

It is recommended that a geotechnical site investigation program be completed in the TMF area during subsequent design studies in order to refine the assumptions made for the PEA cost estimate. The site investigation would include drilling, test pitting, laboratory test work and analysis. This geotechnical program is estimated to cost approximately $600,000 inclusive of all field, laboratory, and engineering costs.

It is also recommended that SMGL investigate acquiring some small portions of additional property for the TMF.

28.7 Environmental

It is recommended that baseline studies and community engagement activities continue as project design advances in order to initiate the environmental assessment process and receive all federal and provincial permits in a timely manner. The cost for completion of all baseline field programs, including soil and terrain assessment for reclamation and closure planning, is estimated to be in the order of $1 million. The environmental assessment and permitting cost, including application for a Schedule 2 amendment of the MMER under the *Fisheries Act*, is estimated to be in the order of $1.5 million.

29 REFERENCES

British Columbia Ministry of the Environment, 2007. Manual of Operation Hydrology in BC.

Hartman, G. and M. Miles, 2001. Assessment of Techniques for Rainbow Trout Transplanting and Habitat Management in British Columbia. Canadian Manuscript Report of Fisheries and Aquatic Sciences No. 2562. Habitat and Enhancement Branch, Fisheries and Oceans Canada.

Knight Piésold, April 2007. Spanish Mountain Project – Preliminary Tailings Storage Facility Alternatives Study. Letter Report (Cont. No. VA07-01585).

Knight Piésold, April 2008. 2007 Environmental Baseline Summary Report (Ref No. VA102 272/2-1 Rev. 0).

Knight Piésold, August 2010. Alternatives Assessment for Waste and Water Management (Ref. No. VA102-2272/5-1 Rev. 0).

Knight Piésold, not yet issued. Concept Development Study (Ref. No. VA102-272/5-2 Rev. 0).

Lustig, Darney, 2006. 2005 Drill Summary with Recommendations for Exploration on the Spanish Mountain Property, 68 pages and appendices.

MEND, 1991. Acid Rock Drainage Prediction Manual. Mine Environment Neutral Drainage Program. Report 1.16.1b.

Montgommery, 2009. Summary Report on 2009 Exploration Activities on The Spanish Mountain Gold Project, 40 pages.

Peatfield, G.R., 2009. Updated Resource Estimation Report on the Spanish Mountain Gold Deposit., 76 pages and appendices.

Price, W., 1997. Draft Guidelines and Recommended Methods for the Prediction of Metal Leaching and Acid Rock Drainage at Minesites in British Columbia. Reclamation Section, British Columbia Ministry of Energy and Mines.

Sinclair and Blackwell, 2002. Applied Mineral Inventory Estimation, 381 pages.

Singh, 2008. Technical Report on the Spanish Mountain Property, 96 pages.

30 CERTIFICATE OF QUALIFIED PERSONS

30.1 Mike Waldegger, P.Geo.

I, Mike Waldegger of Vancouver, British Columbia, as one of the authors of this technical report titled "NI 43-101 Technical Report – Preliminary Economic Assessment for the Spanish Mountain Project, Likely, BC, dated 20 December 2010, do hereby certify that and make the following statements:

- I am an Associate Geologist with AGP Mining Consultants Inc., with a business address at 92 Caplan Avenue, Suite 246, Barrie, Ontario, L4N 0Z7.
- I am a graduate of The University of Ottawa (B.Sc. Hons.) in 1998.
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, Registration #33582.
- I have practiced my profession in the mining industry continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience includes 13 years experience in the mining sector prospecting, managing RC and diamond drill programs, managing data, and estimating resources. I have been involved in numerous projects around the world in both base metals and precious metals deposits.
- I am responsible for the contents of Sections 4 to 15, 17, 21.1, and 28.1 of this technical report titled "NI 43-101 Technical Report – Preliminary Economic Assessment for the Spanish Mountain Project, Likely, BC, dated 20 December 2010."
- I have no prior involvement with the property that is the subject of the Technical Report.
- I visited the property described in this report from the 3rd to the 5th of August, 2010.
- As of the date of this Certificate, to my knowledge, information, and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 20th day of December 2010, at Vancouver, British Columbia.

"Original Document Signed and Sealed"

Michael Waldegger, P.Geo

30.2 Gordon Zurowski, P.Eng.

I, Gordon Zurowski, of Stouffville, Ontario, as one of the authors of this technical report titled "NI 43-101 Technical Report – Preliminary Economic Assessment for the Spanish Mountain Project, Likely, BC, dated 20 December 2010, do hereby certify that and make the following statements:

- I am a Principal Mine Engineer with AGP Mining Consultants Inc., with a business address at 92 Caplan Avenue, Suite 246, Barrie, Ontario, L4N 0Z7.
- I am a graduate of University of Saskatchewan, B.Sc. Geological Engineering, 1989.
- I am a member in good standing of the Association of Professional Engineers of Ontario, Registration #100077750.
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience includes the design and evaluation of open pit mines for the last 20 years.
- I am responsible for the preparation of Sections 1.2, 1.4, 1.9, 1.10, 17.3, 18.2, 23.1.1, 23.1.2, 23.2.1, 23.2.2, 23.2.5, 23.2.7, 24.0, 26.2, and 27.3 of this technical report titled "NI 43-101 Technical Report – Preliminary Economic Assessment for the Spanish Mountain Project, Likely, BC, dated 20 December 2010."
- I have no prior involvement with the property that is the subject of the Technical Report.
- I visited the property described in this report on the 20th and 21st of April.
- As of the date of this Certificate, to my knowledge, information, and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.

Signed and dated this 20th day of December 2010, at Barrie, Ontario.

"Original Document Signed and Sealed"

Gordon Zurowski, P.Eng.

30.3 Andy Holloway, P.Eng.

I, Andy Holloway, P.Eng., of Peterborough, Ontario, as one of the authors of this technical report titled "NI 43-101 Technical Report – Preliminary Economic Assessment for the Spanish Mountain Project, Likely, BC, dated 20 December 2010, do hereby certify that and make the following statements:

- I am a Principal Process Engineer with AGP Mining Consultants Inc. with a business address at 92 Caplan Ave., Ste. #246, Barrie, Ontario, Canada, L4N 0Z7.
- I am a graduate of the University of Newcastle upon Tyne, England, B.Eng. (Hons.), 1989, and I have practiced my profession continuously since then.
- I am a Professional Engineer licensed by Professional Engineers Ontario (Membership Number 100082475).
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to mineral processing and metallurgy includes 19 years experience in the mining sector covering mineral processing, process plant operation, design engineering, and management. I have been involved in numerous projects around the world in both base metals and precious metals deposits.
- I am responsible for the content of Section 16.0 of this technical report titled "NI 43-101 Technical Report – Preliminary Economic Assessment for the Spanish Mountain Project, Likely, BC, dated 20 December 2010."
- I have no prior involvement with the property that is the subject of the Technical Report.
- I have not visited the property described in this report.
- As of the date of this Certificate, to my knowledge, information, and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 20th day of December 2010, at Peterborough, Ontario.

"Original Document Signed and Sealed"

Andy Holloway, P.Eng.

30.4 Mario Colantonio, P.Eng.

I, Mario Colantonio of Timmins, Ontario, as one of the authors of this technical report titled "NI 43-101 Technical Report – Preliminary Economic Assessment for the Spanish Mountain Project, Likely, BC, dated 20 December 2010", do hereby certify that and make the following statements:

- I am an Engineering Manager/Principal with Porcupine Engineering Services Inc. with a business address at #200-81 Shamrock Ave, Suite 200, South Porcupine, ON, P0N 1H0.
- I am a graduate of Queens University, Civil Engineering Program, 1985.
- I am a member in good standing of the Association of Professional Engineers of Ontario, Registration No. 8869554.
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to Project Infrastructure includes the design and construction review of mining and milling infrastructure systems throughout 24 years working as a consulting engineer in the mining industry. I have held the position of engineering manager for consulting engineering firms in Northern Ontario for the last 15 years and in that role have overseen the work of teams of mechanical, electrical, civil and structural engineers on several mine and mill projects.
- I am responsible for the contents of Sections 1.8, 20.0, 24.X, 24.XX, 26.X, and 28.X of this technical report titled "NI 43-101 Technical Report – Preliminary Economic Assessment for the Spanish Mountain Project, Likely, BC, dated 20 December 2010."
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I visited the property described in this report on the 20th of April 2010.
- As of the date of this Certificate, to my knowledge, information, and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.

Signed and dated this 20th day of December 2010, at Timmins, Ontario.

"Original Document Signed and Sealed"

Mario Colantonio, P.Eng.

30.5 H. Warren Newcomen, M.S., P.Eng., P.E.

I, H. Warren Newcomen of Kamloops, British Columbia, as one of the authors of this technical report titled "NI 43-101 Technical Report – Preliminary Economic Assessment for the Spanish Mountain Project, Likely, BC, dated 20 December 2010, do hereby certify that and make the following statements:

- I am an Engineer with BGC Engineering Inc. with a business address at #503-11315 Summit Drive, Kamloops, British Columbia, V2C 5R9.
- I am a graduate of the University of British Columbia and the University of California at Berkeley.
- I am a member in good standing of the Association of Professional Engineers of British Columbia, Registration No. 16123.
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to the Project includes open pit slope and underground designs for the: Cortez Hills Project, Nevada; Donlin Creek Project, Alaska; Galore Creek Project, BC; Golden Bear Project, BC; Goldstrike Mine, Nevada; Palabora Mine, South Africa; New Afton Project, BC; KSM Project, BC., Max Molybdenum Mine, BC.
- I am responsible for the preparation of Sections 1.3, 18.1, 26.2, and 27 of this technical report titled "NI 43-101 Technical Report – Preliminary Economic Assessment for the Spanish Mountain Project, Likely, BC, dated 20 December 2010."
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I visited the property described in this report on the 20th of April 2010.
- As of the date of this Certificate, to my knowledge, information, and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.

Signed and dated this 20th day of December 2010, at Kamloops, British Columbia.

"Original Document Signed and Sealed"

H. Warren Newcomen, M.S., P.Eng., P.E.

30.6 Ken Brouwer, P.Eng.

I, Ken Brouwer of Vancouver, British Columbia, as one of the authors of this technical report titled "NI 43-101 Technical Report – Preliminary Economic Assessment for the Spanish Mountain Project, Likely, BC, dated 20 December 2010, do hereby certify that and make the following statements:

- I am a licensed Professional Engineer (Geotechnical) and I am the Managing Director of Knight Piésold Ltd., with a business address at 1400 – 750 West Pender Street, Vancouver, BC, V6C 2T8.
- I am a graduate of The University of British Columbia, B.A.Sc. (Geological Engineering) 1982 and M.Eng. (Civil Engineering) 1985.
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, Registration # 15,117.
- I have practiced my profession in the mining industry continuously for over 25 years and have worked at Knight Piésold since 1985.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- I have over 25 years experience in site investigations; design, construction, operation, and closure of open pit mines; tailings impoundments; heap leach facilities and water management systems.
- I am responsible for the contents of Sections 1.5.2, 1.9, 5.2, 18.2, 22, 24.15, 24.1.6, 27.5, 28.6, 28.7 of this technical report titled "NI 43-101 Technical Report – Preliminary Economic Assessment for the Spanish Mountain Project, Likely, BC, dated 20 December 2010."
- I have no prior involvement with the property that is the subject of the Technical Report.
- I have visited the property described in this report on the 8th of July 2009.
- As of the date of this Certificate, to my knowledge, information, and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 20th day of December 2010, at Vancouver, British Columbia.

"Original Document Signed and Sealed"

Ken Brouwer, P.Eng.

30.7 Morris Beattie, P.Eng.

I, Morris Beattie of Vancouver, British Columbia, as one of the authors of this technical report titled "NI 43-101 Technical Report – Preliminary Economic Assessment for the Spanish Mountain Project, Likely, BC, dated 20 December 2010, do hereby certify that and make the following statements:

- I am a licensed Professional Engineer (Metallurgical) and I am President of Beattie Consulting Ltd., with a business address at 2938 Celtic Avenue, Vancouver, B.C, V6N 3X7.
- I am a graduate of The University of British Columbia, BASc (Mining and Mineral Process Engineering) 1971 and MASc (Mining and Mineral Process Engineering) 1973 and PhD (Mining and Mineral process Engineering) 1983.
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, Registration # 9,523.
- I have practiced my profession in the mining industry continuously for over 35 years and have worked at Beattie Consulting Ltd since 1992.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- I have over 35 years experience in metallurgical testwork, flowsheet development and project development.
- I am responsible for the contents of Sections 16.1, 16.2 and 16.3 of this technical report titled "NI 43-101 Technical Report – Preliminary Economic Assessment for the Spanish Mountain Project, Likely, BC, dated 20 December 2010."
- I have no prior involvement with the property that is the subject of the Technical Report.
- I have visited the property described in this report in October 2009 and September 2010.
- As of the date of this Certificate, to my knowledge, information, and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am not independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 20th day of December 2010, at Vancouver, British Columbia.

"Original Document Signed and Sealed"

Morris Beattie, P.Eng.

APPENDIX A

Raw Sample Data per Domain
Capped Sample Data per Domain
Composited Sample Data
Diamond Drill Holes

Descriptive Statistics on Raw and Capped Sample Data within High Grade Domains

	hgcontct			hgwack			hglower			hgnorth		
RAW ASSAYS	**AU_GPT**	**AU_GPTCAP**	**LENGTH_INT**	**AU_GPT**	**AU_GPTCAP**	**LENGTH_INT**	**AU_GPT**	**AU_GPTCAP**	**LENGTH_INT**	**AU_GPT**	**AU_GPTCAP**	**LENGTH_INT**
Valid cases	3469	3469	3469	1958	1958	1958	1628	1628	1628	3427	3427	3427
Mean	1.04	1.03	1.51	1.10	0.94	1.49	0.81	0.68	1.49	0.43	0.42	1.54
Std. Deviation	1.82	1.77	0.20	6.58	2.91	0.18	6.23	1.84	0.17	1.51	0.99	0.32
CV	1.75	1.71	0.13	5.98	3.10	0.12	7.68	2.73	0.12	3.47	2.37	0.21
Minimum	0.00	0.00	0.50	0.00	0.00	0.70	0.00	0.00	0.63	0.00	0.00	0.75
1st percentile	0.00	0.00	1.00	0.00	0.00	0.75	0.00	0.00	0.78	0.00	0.00	1.00
5th percentile	0.03	0.03	1.35	0.01	0.01	1.30	0.01	0.01	1.15	0.00	0.00	1.50
10th percentile	0.07	0.07	1.50	0.03	0.03	1.50	0.03	0.03	1.50	0.01	0.01	1.50
25th percentile	0.22	0.22	1.50	0.09	0.09	1.50	0.07	0.07	1.50	0.04	0.04	1.50
Median	0.62	0.62	1.50	0.31	0.31	1.50	0.27	0.27	1.50	0.15	0.15	1.50
75th percentile	1.29	1.29	1.50	0.73	0.73	1.50	0.68	0.68	1.50	0.47	0.47	1.50
90th percentile	2.23	2.23	1.50	1.73	1.73	1.50	1.34	1.34	1.50	1.02	1.02	1.50
95th percentile	3.14	3.14	1.56	3.07	3.07	1.50	2.32	2.32	1.51	1.47	1.47	1.83
99th percentile	6.45	6.45	2.50	13.61	13.61	2.00	7.52	7.52	2.00	3.41	3.41	3.00
Maximum	46.80	40.00	5.00	225.00	40.00	5.00	241.00	30.00	3.00	54.40	20.00	6.68

Descriptive Statistics on Raw and Capped Sample Data within Low Grade Domains

	argupr			greywacke			arglower		
RAW ASSAYS	**AU_GPT**	**AU_GPTCAP**	**LENGTH_INT**	**AU_GPT**	**AU_GPTCAP**	**LENGTH_INT**	**AU_GPT**	**AU_GPTCAP**	**LENGTH_INT**
Valid cases	19485	19485	19485	12650	12650	12650	23169	23169	23169
Mean	0.13	0.12	1.55	0.18	0.17	1.51	0.14	0.14	1.55
Std. Deviation	0.56	0.45	0.31	0.92	0.82	0.17	0.97	0.66	0.31
CV	4.49	3.68	0.20	5.21	4.76	0.11	6.90	4.88	0.20
Minimum	0.00	0.00	0.50	0.00	0.00	0.70	0.00	0.00	0.50
1st percentile	0.00	0.00	1.06	0.00	0.00	1.00	0.00	0.00	1.00
5th percentile	0.00	0.00	1.50	0.00	0.00	1.50	0.00	0.00	1.50
10th percentile	0.01	0.01	1.50	0.01	0.01	1.50	0.01	0.01	1.50
25th percentile	0.01	0.01	1.50	0.01	0.01	1.50	0.01	0.01	1.50
Median	0.03	0.03	1.50	0.03	0.03	1.50	0.03	0.03	1.50
75th percentile	0.12	0.12	1.50	0.11	0.11	1.50	0.09	0.09	1.50
90th percentile	0.28	0.28	1.50	0.33	0.33	1.50	0.26	0.26	1.50
95th percentile	0.44	0.44	2.00	0.58	0.58	1.50	0.50	0.50	2.00
99th percentile	1.28	1.28	3.00	2.06	2.06	2.10	1.57	1.57	3.00
Maximum	39.00	20.00	8.50	31.30	20.00	5.11	96.20	30.00	8.53

Descriptive Statistics on 2.5 m Composited Sample Data within the High Grade Domains

	hgcontct			hgwack			hglower			hgnorth		
2.5M COMPOSITES	**AU_GPT**	**AU_GPTCAP**	**LENGTH_INT**	**AU_GPT**	**AU_GPTCAP**	**LENGTH_INT**	**AU_GPT**	**AU_GPTCAP**	**LENGTH_INT**	**AU_GPT**	**AU_GPTCAP**	**LENGTH_INT**
Valid cases	2093	2093	2093	1168	1168	1168	968	968	968	2124	2124	2124
Mean	1.03	1.02	2.50	0.98	0.88	2.51	0.79	0.65	2.50	0.41	0.40	2.50
Std. Deviation	1.28	1.25	0.22	3.01	1.77	0.20	3.68	1.29	0.14	0.78	0.68	0.12
CV	1.24	1.22	0.09	3.09	2.01	0.08	4.68	1.98	0.05	1.89	1.69	0.05
Minimum	0.00	0.00	1.24	0.00	0.00	1.38	0.00	0.00	1.30	0.00	0.00	1.27
1st percentile	0.01	0.01	1.50	0.00	0.00	1.50	0.01	0.01	1.73	0.00	0.00	2.00
5th percentile	0.06	0.06	2.50	0.03	0.03	2.50	0.02	0.02	2.50	0.01	0.01	2.50
10th percentile	0.12	0.12	2.50	0.05	0.05	2.50	0.05	0.05	2.50	0.03	0.03	2.50
25th percentile	0.33	0.33	2.50	0.16	0.16	2.50	0.11	0.11	2.50	0.07	0.07	2.50
Median	0.72	0.72	2.50	0.40	0.40	2.50	0.34	0.34	2.50	0.19	0.19	2.50
75th percentile	1.28	1.28	2.50	0.85	0.85	2.50	0.69	0.69	2.50	0.51	0.51	2.50
90th percentile	2.12	2.12	2.50	1.76	1.76	2.50	1.40	1.40	2.50	0.91	0.91	2.50
95th percentile	2.89	2.89	2.50	3.30	3.30	2.50	2.18	2.18	2.50	1.40	1.40	2.50
99th percentile	5.81	5.81	3.50	11.20	9.54	3.50	5.50	5.50	3.00	3.14	3.09	2.50
Maximum	21.61	18.65	3.73	67.49	19.80	3.74	96.18	18.14	3.50	15.56	12.54	3.71

Descriptive Statistics on 2.5 m Composited Sample Data within the Low Grade Domains

	argupr			greywacke			arglower		
2.5M COMPOSITES	**AU_GPT**	**AU_GPTCAP**	**LENGTH_INT**	**AU_GPT**	**AU_GPTCAP**	**LENGTH_INT**	**AU_GPT**	**AU_GPTCAP**	**LENGTH_INT**
Valid cases	12178	12178	12178	7663	7663	7663	14413	14413	14413
Mean	0.12	0.12	2.50	0.17	0.17	2.50	0.14	0.13	2.50
Std. Deviation	0.35	0.32	0.14	0.58	0.52	0.17	0.70	0.49	0.10
CV	2.85	2.63	0.05	3.46	3.13	0.07	5.10	3.68	0.04
Minimum	0.00	0.00	1.25	0.00	0.00	1.26	0.00	0.00	1.25
1st percentile	0.00	0.00	1.86	0.00	0.00	1.79	0.00	0.00	2.49
5th percentile	0.01	0.01	2.50	0.00	0.00	2.50	0.01	0.01	2.50
10th percentile	0.01	0.01	2.50	0.01	0.01	2.50	0.01	0.01	2.50
25th percentile	0.01	0.01	2.50	0.01	0.01	2.50	0.01	0.01	2.50
Median	0.03	0.03	2.50	0.04	0.04	2.50	0.03	0.03	2.50
75th percentile	0.14	0.14	2.50	0.14	0.14	2.50	0.11	0.11	2.50
90th percentile	0.29	0.29	2.50	0.35	0.35	2.50	0.28	0.28	2.50
95th percentile	0.44	0.44	2.50	0.61	0.61	2.50	0.50	0.50	2.50
99th percentile	1.21	1.21	2.92	2.11	2.10	3.38	1.43	1.43	2.50
Maximum	15.05	11.05	3.74	16.01	11.99	3.73	57.81	26.23	3.74

Drill Holes Used in Resource Estimate

05-DDH-251	06-DDH-503	06-DDH-556	07-DDH-614	07-DDH-687	08-DDH-739	08-DDH-796	09-DDH-852	ROG-009
05-DDH-252	06-DDH-504	06-DDH-557	07-DDH-618	07-DDH-688	08-DDH-740	08-DDH-797	09-DDH-853	ROG-010
05-DDH-253	06-DDH-505	06-DDH-558	07-DDH-621	07-DDH-689	08-DDH-741	08-DDH-798	09-DDH-854	ROG-011
05-DDH-254	06-DDH-506	06-DDH-559	07-DDH-623	07-DDH-690	08-DDH-742	08-DDH-799	09-DDH-856	ROG-012
05-DDH-255	06-DDH-507	06-DDH-560	07-DDH-627	07-DDH-691	08-DDH-743	08-DDH-800	09-DDH-857	ROG-013
05-DDH-256	06-DDH-508	06-DDH-561	07-DDH-628	07-DDH-692	08-DDH-744	08-DDH-801	09-DDH-858	ROG-014
05-DDH-257	06-DDH-509	06-DDH-562	07-DDH-629	07-DDH-693	08-DDH-745	08-DDH-802	09-DDH-859	ROG-015
05-DDH-258	06-DDH-510	06-DDH-563	07-DDH-634	07-DDH-694	08-DDH-746	08-DDH-803	09-DDH-860	ROG-016
05-DDH-259	06-DDH-511	06-DDH-564	07-DDH-635	07-DDH-695	08-DDH-747	08-DDH-804	09-DDH-861	ROG-017
05-DDH-260	06-DDH-512	06-DDH-565	07-DDH-637	08-DDH-696	08-DDH-748	08-DDH-805	09-DDH-862	ROG-018
05-DDH-261	06-DDH-513	06-DDH-566	07-DDH-638	08-DDH-697	08-DDH-749	08-DDH-806	09-DDH-865	
05-DDH-263	06-DDH-514	06-DDH-567	07-DDH-640	08-DDH-698	08-DDH-750	08-DDH-807	09-DDH-866	
05-DDH-264	06-DDH-515	06-DDH-568	07-DDH-642	08-DDH-699	08-DDH-751	08-DDH-808	09-DDH-867	
05-DDH-265	06-DDH-516	06-DDH-569	07-DDH-643	08-DDH-700	08-DDH-752	08-DDH-809	09-DDH-868	
05-DDH-266	06-DDH-517	06-DDH-570	07-DDH-644	08-DDH-701	08-DDH-753	08-DDH-810	09-DDH-869	
05-DDH-267	06-DDH-519	06-DDH-571	07-DDH-645	08-DDH-702	08-DDH-754	08-DDH-811	09-DDH-870	
05-DDH-268	06-DDH-520	06-DDH-572	07-DDH-646	08-DDH-703	08-DDH-755	08-DDH-812	09-DDH-871	
05-DDH-269	06-DDH-521	07-DDH-573	07-DDH-647	08-DDH-704	08-DDH-756	08-DDH-813	09-DDH-872	
05-DDH-270	06-DDH-522	07-DDH-574	07-DDH-648	08-DDH-705	08-DDH-757	08-DDH-814	09-DDH-873	
05-DDH-271	06-DDH-523	07-DDH-576	07-DDH-649	08-DDH-706	08-DDH-758	08-DDH-815	09-DDH-874	
05-DDH-272	06-DDH-524	07-DDH-577	07-DDH-650	08-DDH-707	08-DDH-759	08-DDH-816	09-DDH-875	
05-DDH-273	06-DDH-525	07-DDH-578	07-DDH-651	08-DDH-708	08-DDH-760	08-DDH-817	09-DDH-876	
05-DDH-274	06-DDH-526	07-DDH-579	07-DDH-652	08-DDH-709	08-DDH-761	08-DDH-818	09-DDH-877	
05-DDH-275	06-DDH-527	07-DDH-580	07-DDH-653	08-DDH-710	08-DDH-762	08-DDH-819	09-DDH-878	
05-DDH-276	06-DDH-528	07-DDH-581	07-DDH-654	08-DDH-711	08-DDH-763	08-DDH-820	09-DDH-879	
05-DDH-277	06-DDH-529	07-DDH-582	07-DDH-655	08-DDH-712	08-DDH-764	08-DDH-821	09-DDH-880	
05-DDH-278	06-DDH-530	07-DDH-583	07-DDH-656	08-DDH-713	08-DDH-765	08-DDH-822	09-DDH-881	
05-DDH-279	06-DDH-531	07-DDH-584	07-DDH-657	08-DDH-714	08-DDH-766	08-DDH-823	09-DDH-882	
05-DDH-280	06-DDH-532	07-DDH-585	07-DDH-658	08-DDH-715	08-DDH-767	08-DDH-824	09-DDH-883	
05-DDH-281	06-DDH-533	07-DDH-586	07-DDH-659	08-DDH-716	08-DDH-768	08-DDH-825	09-DDH-884	
05-DDH-282	06-DDH-534	07-DDH-587	07-DDH-660	08-DDH-717	08-DDH-772	08-DDH-826	09-DDH-885	
05-DDH-283	06-DDH-535	07-DDH-588	07-DDH-661	08-DDH-718	08-DDH-773	08-DDH-827	09-DDH-886	
05-DDH-284	06-DDH-536	07-DDH-590	07-DDH-662	08-DDH-719	08-DDH-774	08-DDH-828	09-DDH-887	
05-DDH-285	06-DDH-537	07-DDH-591	07-DDH-663	08-DDH-720	08-DDH-775	08-DDH-829	09-DDH-888	
06-DDH-286	06-DDH-538	07-DDH-592	07-DDH-664	08-DDH-721	08-DDH-776	08-DDH-830	09-DDH-889	
06-DDH-287	06-DDH-539	07-DDH-593	07-DDH-665	08-DDH-722	08-DDH-778	08-DDH-831	09-DDH-890	
06-DDH-288	06-DDH-540	07-DDH-594	07-DDH-666	08-DDH-723	08-DDH-779	08-DDH-832	09-DDH-891	
06-DDH-289	06-DDH-541	07-DDH-595	07-DDH-667	08-DDH-724	08-DDH-781	08-DDH-833	09-DDH-892	
06-DDH-290	06-DDH-542	07-DDH-596	07-DDH-668	08-DDH-725	08-DDH-782	08-DDH-834	09-DDH-893	
06-DDH-291	06-DDH-543	07-DDH-597	07-DDH-669	08-DDH-726	08-DDH-783	08-DDH-835	09-DDH-894	
06-DDH-292	06-DDH-544	07-DDH-597A	07-DDH-670	08-DDH-727	08-DDH-784	08-DDH-836	09-DDH-895	
06-DDH-293	06-DDH-545	07-DDH-598	07-DDH-671	08-DDH-728	08-DDH-785	09-DDH-838	09-DDH-896	
06-DDH-294	06-DDH-546	07-DDH-598A	07-DDH-672	08-DDH-729	08-DDH-786	09-DDH-839	09-DDH-897	
06-DDH-295	06-DDH-547	07-DDH-599	07-DDH-673	08-DDH-730	08-DDH-787	09-DDH-840	09-DDH-898	
06-DDH-296	06-DDH-548	07-DDH-600	07-DDH-674	08-DDH-731	08-DDH-788	09-DDH-841	ROG-001	
06-DDH-297	06-DDH-549	07-DDH-601	07-DDH-676	08-DDH-732	08-DDH-789	09-DDH-842	ROG-002	
06-DDH-298	06-DDH-550	07-DDH-602	07-DDH-677	08-DDH-733	08-DDH-790	09-DDH-843	ROG-003	
06-DDH-299	06-DDH-551	07-DDH-603	07-DDH-680	08-DDH-734	08-DDH-791	09-DDH-844	ROG-004	
06-DDH-300	06-DDH-552	07-DDH-604	07-DDH-683	08-DDH-735	08-DDH-792	09-DDH-845	ROG-005	
06-DDH-500	06-DDH-553	07-DDH-606	07-DDH-684	08-DDH-736	08-DDH-793	09-DDH-849	ROG-006	
06-DDH-501	06-DDH-554	07-DDH-607	07-DDH-685	08-DDH-737	08-DDH-794	09-DDH-850	ROG-007	
06-DDH-502	06-DDH-555	07-DDH-611	07-DDH-686	08-DDH-738	08-DDH-795	09-DDH-851	ROG-008	

APPENDIX B

Process Plant Design Criteria
Equipment List
Plant Layout

PROCESS DESIGN CRITERIA

SPANISH MOUNTAIN PROJECT

Revision: E -- For Report, December 2010
For: PEA Preliminary Design

Criteria		Units	Design Data	Source	Comments
1. Project Fundamentals					
Mean Throughput	Annual	dmtpa	9,025,000	1	
	Monthly	dmtpm	752,083	1	
	Daily	**dmtpd**	**25,000**	1	
2. Operating Schedule					
2.1 Crushing circuit					
Hours per Shift			12	1	
Shifts per Day			2	1	
Days per Week			7	1	
Operating Days per Annum			361	2	*Annual Shutdown*
Availability		%	86.0	2	*typical*
Operating Hours per Annum		h	7,451	4	*Derived*
Nominal feedrate for Crusher Circuit		dmtph	1,211	4	
		dmtpd	25,000	4	*Derived*
2.2 Mill					
Hours per Shift		h	12	1	
Shifts per Day		h	2	1	
Days per Week		d	7	1	
Days per annum		d	361	2	*Allow for Annual Shutdown*
Availability		%	92.5	1	
Operating Hours per Annum		h	8,014	4	*Derived*
Nominal feedrate for Mill Circuit		dmtph	1126.1	4	
		dmtpd	25,000	4	*at quoted availability*
3. ROM Physical Characteristics					
Moisture Range					
Min		%	2	2	
Design		%	5	2	
Max		%	8	2	*Rains*
Solid Particle Density (Ore)		t/m^3	2.76	2	
	(waste)	t/m^3	2.6	2	
	(average)	t/m^3	2.72	2	
Bulk Density					
As Mined (Coarse)		t/m^3	1.64	2	
Mill Feed (Fine)		t/m^3	2.1	2	
Abrasion Index (Bond AI)		g	0.22		*From Testwork*
Rod Mill WI (Bond) - Metric		kWh/t	14.6		
Ball Mill WI (Bond) - Metric					
100 mesh		kWh/t	13.7	3	*Average*
Material Handling					

Criteria	Units	Design Data	Source	Comments
Angle of Repose	Deg	45	2	*Typical*
Drawdown Angle	Deg	55	2	*Typical*
4. ROM Chemical Characteristics				
Head Grade (Mill Feed)				
Au	g/t	0.67	1	
C_{total}	%	2.93	1	
S_{total}	%	1.92	1	
5. Crushing Plant				
Method	Single gyratory, discharging into surge pocket			
Haul Truck Capacity	t	175	2	
Haul Truck Max Frequency	# per h	9	2	
Truck turn around @ Max Freq.	min	7	4	*time to reverse in, dump and clear tip*
Ore Delivery Rate				
Average	dmtph	1,211	4	
Peak	dmtph	1,575	4	*design bin for expected peak*
Surge Bin Capacity	min	15	4	
	dmt	394	4	*equivalent to 2 trucks at peak rate*
Crusher				
Design Throughput	tph	1,284	4	*Average, plus 6% SF*
Average Throughput	tph	1,211	4	*derived*
Size/Type	Metso Superior 54-75			*from topsize and capacity charts*
Feed Opening	mm	1,370	4	*from topsize and capacity charts*
OSS	mm	160	4	*discharge opening*
Size Distributions				
Feed F_{100}	mm	1,000	2	*40" Max feed size = 80% of opening*
Feed F_{80}	mm	600	2	
Product P_{100}	mm	150	2	
Product P_{80}	mm	125	2	
Apron Feeder				
Peak Capacity	dmtph	1,575	2	
Design Capacity	dmtph	1,800	2	*10% over max*
Feeder Width	mm	2,000	2	*72" x 30' approx*
Conveyors				
Peak Capacity	dmtph	1,575	2	
Design Capacity	dmtph	1,800	2	*to match feeder*
Belt Width	mm	1,375	2	*54" belt, 2m/s(max)*
6. Mill Feed Stockpile				
Live Storage Capacity Required	h	18	4	
	dmt	20,270	2	
	m^3	12,360	4	
Feeder Type	Triple Apron Feeders			*Changed to 3*

Criteria	Units	Design Data	Source	Comments
Arrangement	In Line			
Feeder Capacity Peak	dmtph	1000.0	5	*normally 2 runnin. VSD*
7. Grinding Circuit				
Arrangement	SAG mill, closed circuit grate discharge Ball Mill			
Mill Feed Conveyor Design				
Peak Tonnage	dmtph	1239	2	*average mill feed +10%*
Design Tonnage	dmtph	1400	4	*average mill feed +25%*
Mill Circuit Throughput				
Average	dmtph	1126	2	
Design	dmtph	1200	4	
Material Size				
Feed Size (F_{100})	mm	150	2	
Feed Size (F_{80})	mm	125	2	
Final Grind Size (P_{80})	um	180	2	
7.1 SAG Mill				
Mill Diameter	ft	28	2	
	m	8.54		
Mill EGL	ft	18	2	
	m	5.5		
Applied Power @ Pinion	kWh/t	6.7	2	
Pinion Power Consumption	kW	5200	2	
Recommended Installed Power	kW	5600	2	
	Hp	7500		
Mill Speed	%Cs	70	2	
=>	rpm	10.0	2	
Charge Volume				
Nominal	%	20	2	
Maximum	%	25	2	
Media Volume				
Nominal	%	5	2	
Maximum	%	15	2	
Trommel Screen Size	mm	40	2	
Trommel Oversize	%	7	2	
Mill Discharge % Solids	%	72	2	
7.1.1 Grinding Media & Liners				
Media Type	Chrome Balls, 125mm			
Rate of Consumption	kg/t	0.10	2	
	tpa	903	4	
Top-up Size	mm	125	2	
Liner Type	Cast Mn, wave			
Rate of Consumption	kg/t	0.10	2	
	tpa	902.5	4	

Criteria	Units	Design Data	Source	Comments
7.1.2 Classification Screen				*TBA*
Screen Type	Inclined Deck, slotted Screens			
Screen Width	ft	12	2	
	m	3.7		
Screen Length	ft	20	2	
	m	6.1		
Angle	deg	5.0		*inclined to improve capacity.*
Slot Width	mm	4.0		
Slot Length	mm	15.0		
Screen Oversize	%	14.0	2	*Assumed*
Screen Oversize % solids	%	85.0	2	*Estimated*
7.2 Ball Mill				
Mill Diameter	ft	22.5	2	
	m	6.86	4	*derived*
Mill EGL	ft	33	2	
	m	10.1	4	*derived*
Applied Power @ Pinion	kWh/t	6.7	2	
Pinion Power Consumption	kW	6800	2	
Recommended Installed Power	kW	7500	2	
	Hp	10000		
Mill Speed	%Cs	70	2	
=>	rpm	12.5	2	
Media Volume				
Nominal	%	30	2	
Maximum	%	32	2	
Discharge Screen Slots	mm x mm	-	2	*Not Installed - overflow*
Trommel Screen Slots	mm x mm	10x15		
Mill Discharge % Solids	%	70	2	
7.2.1 Grinding Media & Liners				
Media Type	Hi Chrome Balls, 38mm			
Rate of Consumption	kg/t	0.18	4	
	tpa	1624.5	4	
Top-up Size	mm	50	2	
Liner Type	Rubber, polymet ends			
Rate of Consumption	kg/t	0.10	2	
	tpa	902.5	4	
7.2.2 Hydrocyclones	Five Cyclone Cluster - 840mm			
Feed Flow				
Nominal	m^3/h	4788.9	4	
Maximum	m^3/h	5746.7	4	
Operating Pressure				

Criteria	Units	Design Data	Source	Comments
Nominal	kPa	70	2	
Maximum	kPa	80	2	
Circulating Load				
Nominal	%	230	4	
Maximum	%	250	4	
Underflow % solids				
Nominal	%	70	4	
Maximum	%	75	4	
Underflow Mass Split				
Nominal	%	69.7	2	
Maximum	%	71.4	2	
Overflow % solids				
Nominal	%	32	2	
Maximum	%	34	2	
Nominal Overflow P_{80}	um	184	3	
7.2.2 Gravity Concentration	Knelson KC-XD48			
Feed Stream	Cyclone Underflow			
Mass Split	%	20	2	
Mass Feed Rate				
Nominal	t/h	518	4	
Maximum	t/h	650	4	
Volume Feed Rate				
Nominal	m^3/h	389.1	4	
Maximum	m^3/h	488.2	4	
Water Consumption	m^3/h	50	5	
Concentrate Recovery	%	1.00	2	
Concentrate Pulp Density	%	65	2	
9. Flotation Circuit				
Description	Rougher Flotation followed by 2 stages of cleaning			
9.1 Rougher Flotation				
Bulk Circuit Feed Flow	dry t/h	1126	4	
	m^3/h	2755	2	*from mass balance*
Rougher Res. Time Req.	min	20	3	*8 min x 2.5 Scale up*
Total Active Volume req.	m^3	918.3	4	
Configuration	Tank cells in series 1 x 6			
No of Cells Total	#	6	2	
Selected Cells	Metso RCS 160			
per cell volume	m^3	180	5	
Active Volume	%	85	5	
Installed Volume	m^3	918	4	
Installed Residence Time	min	20.0	4	

Criteria	Units	Design Data	Source	Comments
Rougher Mass Pull				
Min	%	12	2	
Max	%	16	2	
Design	%	13	2	
Rougher Concentrate % solids				
Min	%	20	2	
Max	%	25	2	
Design	%	22	2	
Rougher Concentrate solids Density	t/m^3	3.1	2	*assumed*
Rougher Concentrate Froth Factor		3.5	2	*Average*
Approx Air Requirement				
Flow	m^3/min	180	5	
Pressure	kPa	54	5	
9.2 Cleaner Flotation				
Cleaner 1 Calc Feed Flow	m^3/h	614.7	4	*@ 20% solids*
Design Flow	m^3/h	700.0	2	
Cleaner 1 Res. Time Req.	min	14	3	
Total Active Volume req.	m^3	163.3	4	
Configuration	8 tanks in series			*6 min to avoid short-circuiting*
No of Cells Total	#	8	2	
Selected Cells	Metso RCS 20			
per cell volume	m^3	24	5	*vendor info*
Cell Step Height	mm	300	2	
Active Volume	%	85	2,5	*air holdup, mechanism*
Installed Active Volume	m^3	163.2	4	
Installed Residence Time	min	14.0	4	
Cleaner 1 Unit Mass Pull				
Min	%	65	2	
Max	%	75	2	
Design	%	70	3	
Cleaner 1 Concentrate % solids				
Min	%	10	2	
Max	%	20	2	
Design	%	17	3	
Cleaner 1 Concentrate solids Density	t/m^3	3.1	2	*Deliverable of next pilot campaign*
Cleaner 1 Concentrate Froth Factor		3.5	2	
Approx Air Requirement				
Flow	m^3/min	64	5	
Pressure	kPa	27	5	
9.3 Regrind Mill				
Fresh Feed Rate (Expected)	tph	101	4	*@ 9% mass pull*

Criteria	Units	Design Data	Source	Comments
(Design)	tph	120		
Feed Size Distribution				
Feed F_{100}	um	180	2	
Feed F_{80}	um	125	2	
Product Size Distribution				
Circuit P_{100}	um	48	2	
Circuit P_{80}	um	20	2	
Circuit Description	Metso Tower Mill with classifier			
Mill Installed Size				
Calc Applied Power	kW	1,600	2	
Mill Installed Power	kW	1,650	2	
Design applied power req.	kWh/t	13.33	2	*per tonne of concentrate*
7.2.2 Hydrocyclones	Five Cyclone Cluster - 375mm			*Linatex*
Feed Flow				
Nominal	m^3/h	740.0	2	
Maximum	m^3/h	888.0	4	*+20%*
Operating Pressure				
Nominal	kPa	70	2	
Maximum	kPa	80	2	
Circulating Load				
Nominal	%	200	4	
Maximum	%	250	4	
Underflow % solids				
Nominal	%	65	4	
Maximum	%	70	4	
Underflow Mass Split				
Nominal	%	66.7	2	
Maximum	%	71.4	2	
Overflow % solids				
Nominal	%	0	2	
Maximum	%	0	2	
Nominal Overflow P_{80}	um	20	2	
9.4 Flotation Air Blowers				
Air requirement				
Roughers	m^3/min	180	4	
Cleaners	m^3/min	64	4	
10. Concentrate Handling				
10.1 CIL Preleach Thickener				
Description	Outotec Supaflow			
Duty	High Rate			
Thickener Fresh Feed				
Dry feedrate	dmtph	120	2	*10% masspull*

Criteria	Units	Design Data	Source	Comments
Flow	m^3/h	521.7	2	
Density	kg/l	1.15	2	
% Solids	%	20.0	2	
Thickener U/F density				
% Solids	%	55	2	
Flow	m^3/h	122	2	
Selected Thickener				
Diameter	m	20.0	2	
Area	m^2	314.2	4	
Thickener Unit Area	$t/d/m^2$	7.8	4	
Thickener Rise Rate	m/h	1.27	4	
11. CIL Circuit				
Description	Pre-Aeration followed by Conventional CIL, 8 tanks total			
Availability	%	95.8	2	
Availability	h/d	23	4	
Feed				
Dry feed rate	t/hr	107.2	4	*from mass balance*
		2465.3		
Flow	m^3/hr	122.3	4	
Density	kg/l	1.59	2	
% Solids	%	55.0	2	
Au	g/t	6.50	2	*from mass balance*
Areation Time Required	h	3	2	
Tank Working Volume	m^3	600	2	
Number of tanks installed	#	1	2	
Total Capacity				
Volume	m^3	600	4	
Time	h	4.9	4	
Tank pH		10.5	2	
Air flow rate	m3/s		2	
Leach Residence Time Required	h	24	2	
Leach Tank Working Volume	m^3	500	2	
Number of tanks installed	#	6	2	
Total Capacity				
Volume	m^3	3,000	4	
Time	h	24.5	4	
Tank pH		10.5	2	
Air flow rate	m3/s		2	
NaCN concentration				
NaCN consumption	g/t	400.0	2	

Criteria	Units	Design Data	Source	Comments
Au Extraction (stage)	%	95	2	changed to 95%
	g/day	15223	4	changed to 23h
Carbon Details (GAC)				
Design GAC Concentration	gpl	22	2	
GAC Inventory in Leach Tanks	t	66	4	
Anticipated Carbon Loading				
Au	g/t	2000	2	
Stripped Carbon Loading				
Au	g/t	100	2	
Carbon cycle	t/d	8.0	4	
GAC Transfer Pumps Volume (pushing)				*Metso vert spindle pumps*
Hours per day pumping	h/d		2	
Slurry flowrate @ design Conc	m3/h		2	
11. Elution				
Description	Zadra Method			
Tailings Slurry				
Elution Capacity	t/d	10.0	2	
12. Cyanide Destruction				
Description				
Tailings Slurry				
Tailings Flow Rate	m^3/h	122.3	4	
Tailings Density	kg/l	1.50	2	
NaCN concentration	ppm	50.00	2	
NaCN Destruction Method	Inco SO2/air			
# of stages	2			
tank residence time	minutes	15		
tank volume	m3	30.6		*40m3 total volume*
Total CN-destroyed	g/hr	3119.2		
Metabisulfite consumed	g/hr	37040.9		
Copper sulfate concentration	mg/l	20.0		
Copper sulfate consumption	g/hr	2445.5		
NaCN discharge concentration	ppm	<10		
13. Process Water				
Description				
Process Water Requirement				
Process Water - SAG Mill	m^3/h	810	4	
Process Water - Ball Mill	m^3/h	1475	4	

Criteria	Units	Design Data	Source	Comments
Process Water - Rougher Float	m^3/h	56	4	
Process Water - Cleaner Flotation	m^3/h	4	4	
Process Water - Reagents 1	m^3/h	10	4	
Process Water Total	m^3/h	2355	4	
Average Return Water Recovery	m^3/h	434.1	4	
Return Water Make up Water Requirement	m^3/h	1926.9	4	
14. Reagents				
14.1 Frother				
General				
Name	Methylisobutyl Carbinol (MIBC)			
Transported By	supplier name			
Transport Form	Liquid			
Transport Packaging	200 L drums on pallets			
Off-loading method	Forklift			
Solution Density	kg/l	0.82	5	
pH		n/a		
Consumption				
Average	g/t	45	3	*from testwork*
Design	g/t	50	2	*average +10%*
On Site Storage (Max)	m3	2.4	2	
	litres	2,400	4	
	kg	1,968	4	
Mixed Concentration	% w/w	100	2	
Storage				
Storage Capacity	days	1.57	4	
Volume	m^3	2.40	2	
Head Tank				
Volume	litres	2,000	2	
Storage Capacity	days	1.31	4	
14.2 Bulk Collector				
General				
Name	Potassium Amyl Xanthate			
Transported By	supplier name			
Transport Form	Pellets			
Transport Packaging	1 tonne bulk bags			
Off-loading method	Flatbed. Offload with forklift			
pH		n/a		
Consumption				
Average	g/t	90	3	*from testwork*
Design	g/t	100	4	*average +10%*
On Site Storage (Max)	t	4.0	2	

Criteria	Units	Design Data	Source	Comments
(Days)	d	1.8	4	
Mixed Concentration	% w/w	10	2	
Storage				
Mix Tank Capacity	days	0.32	4	
Volume	m^3	8.00	2	
Head Tank				
Volume	m3	5.00	4	
Storage Capacity	days	0.20	2	
14.3 pH Modifier				
General				
Type	Lime			
Name				
Transported By	Road			
Transport Form	bulk delivery			
Off-loading method	vacuum line			
Consumption				
Average	g/t	70	2	
Design	g/t	80	4	*average +10%*
On Site Storage				
Silo	kg	10,000	2	
(Days)	d	5.0	4	
Mixed Concentration	% w/w	10.00	2	
Storage				
Mix Tank Capacity	days	0.50	4	
Volume	m^3	10.0	2	
Head Tank				
Volume	m^3	8.0	4	
Storage Capacity	days	0.40	2	
14.4 Flocculant (Concentrate Thickener)				
General				
Type	Magnafloc 333			
Name				
Transported By				
Transport Form	Powder			
Transport Packaging	1 tonne bulk bags			
Off-loading method	Forklift. 1 bag per skid			
Consumption				
Average	g/t	10	2	*estimated*
Design	g/t	11	4	*average +10%*

Criteria	Units	Design Data	Source	Comments
On Site Storage				
Max Mass	t	2	2	
(Days)	d	6.9	4	
Mixed Concentration	% w/w	1.00	2	
Storage				
Storage Capacity	days	0.36	4	
Volume	m^3	10.00	2	
Head Tank				
Volume	m^3	8.00	2	
Storage Capacity	days	0.29	4	
14.5 NaCN				
General				
Name	Sodium Cyanide			
Transported By	supplier name			
Transport Form	Powder			
Transport Packaging	1000 kg bags			
Off-loading method	Flatbed. Offload with forklift			
pH		n/a		
Consumption				
Average	g/t	400	3	
Design	g/t	440	4	*average +10%*
On Site Storage (Max)	t	20.0	2	
(Days)	d	2.0	4	
Mixed Concentration	% w/w	10	2	
Storage				
Mix Tank Capacity	days	0.27	4	
Volume	m^3	30.00	2	
Head Tank				
Volume	m3	20.00	4	
Storage Capacity	days	0.18	2	
14.6 NaOH				
General				
Name	Sodium Hydroxide			
Transported By	supplier name			
Transport Form	Liquid			
Transport Packaging	1m3 totes			
Off-loading method	Flatbed. Offload with forklift			
pH		n/a		
Consumption				
Average	g/t	4	3	
Design	g/t	5	4	*average +10%*

Criteria	Units	Design Data	Source	Comments
On Site Storage (Max)	t	2.0	2	
(Days)	d	20.0	4	
Mixed Concentration	% w/w	100	2	
<u>14.7 HCl</u>				
General				
Name	Hydrochloric Acid			
Transported By	supplier name			
Transport Form	Liquid			
Transport Packaging	1m3 totes			
Off-loading method	Flatbed. Offload with forklift			
pH		n/a		
Consumption				
Average	g/t	14	3	
Design	g/t	16	4	*average +10%*
On Site Storage (Max)	t	2.0	2	
(Days)	d	5.7	4	
Mixed Concentration	% w/w	100	2	
<u>14.8 Na2S2O3</u>				
General				
Name	Sodium Metabisulfite			
Transported By	supplier name			
Transport Form	Powder			
Transport Packaging	25 kg bags			
Off-loading method	Flatbed. Offload with forklift			
pH		n/a		
Consumption				
Average	g/t	36	3	
Design	g/t	40	4	*average +10%*
On Site Storage (Max)	t	4.0	2	
(Days)	d	4.5	4	
Mixed Concentration	% w/w	10	2	
Storage				
Mix Tank Capacity	days	0.80	4	
Volume	m^3	8.00	2	
Head Tank				
Volume	m3	5.00	4	
Storage Capacity	days	0.50	2	

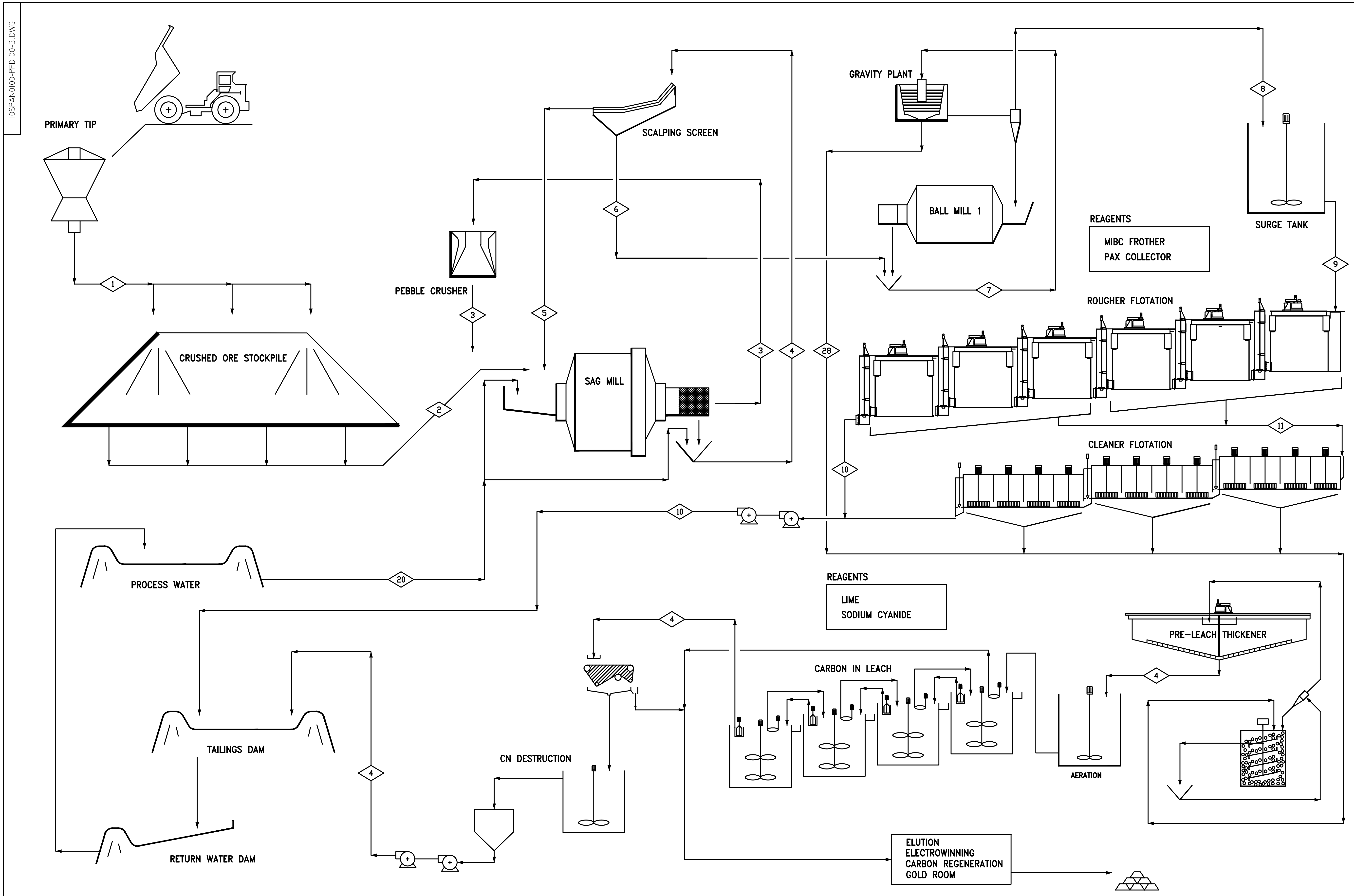
Criteria	Units	Design Data	Source	Comments
14.9 CuSO4				
General				
Name	Copper Sulfate			
Transported By	supplier name			
Transport Form	Powder			
Transport Packaging	25 kg bags			
Off-loading method	Flatbed. Offload with forklift			
pH		n/a		
Consumption				
Average	g/t	2.3	3	
Design	g/t	3	4	*average +10%*
On Site Storage (Max)	t	1.0	2	
(Days)	d	17.0	4	
Mixed Concentration	% w/w	10	2	
Storage				
Mix Tank Capacity	days	6.67	4	
Volume	m^3	5.00	2	
Head Tank				
Volume	m3	5.00	4	
Storage Capacity	days	6.67	2	

Data Source:

Information by client	1
Assumed by Engineer	2
From Testwork	3
Derived from other Data	4
From Vendor	5

Project:	Spanish Mountain Gold Project	Revision:	0
Ref:	09SPAN0100	Issue Date:	03-Dec-10
Document:	Mechanical Equipment List - Mill Area	For:	Comment and Preliminary Review
		Changes:	as hilighted

Area	Item	No	Type	Description	Specification	kW	SBY	Notes
100	CGA	04	M	Primary Crusher	Supplier: Metso Type: Superior 42-65	350		Soft Start reqd.
100	FCV	32	CON	Stockpile Feed Conveyor	Supplier: Continental Type:	225		Max. 12 degree angle to stockpile.
100	FCV	52	CON	SAG Mill Feed Conveyor	Supplier: Continental Type:	110		
200	MSA	16	M	SAG mill	Supplier: Outotec Type:	5700		
200	CCB	46	M	Pebble Crusher	Supplier: Metso Type: HP200	132		
200	PCB	58	M	SAG Mill discharge Pump	Supplier: Metso Type: XM400	150		all metal pumps
200	SVE	64	M	Scalping Screen	Supplier: Metso Type: 12' x 20' SD	55		4mm SWF
200	XLC	78	M	Mill Area Overhead Crane	Supplier: Type: 20t capacity	30		total kW consists of short/long travel & hoist. Sized to lift motor
200	MZA	82	M	SAG Mill Liner Handler	Supplier: RME Type: 7-Axis	11		
220	MBA	02	M	Ball Mill	Supplier: Outotec Type:	7500		
220	PCA	08	M	Ball Mill Discharge Pump A	Supplier: Metso Type: XR 350	300		
220	PCA	09	M	Ball Mill Discharge Pump B	Supplier: Metso Type: XR 350	300		
220	YAA	12	M	Ball Mill Cyclone Cluster	Supplier: Linatex Size: 10 x 840 mm	-		9 operating, 1 standby
220	GDA	18	M	Centrifugal Concentrator	Supplier: Knelson Type: XD-70	150		
300	XCB	02	M	Rougher Flot Cell #1	Supplier: Metso Type: RCS 160	160		c/w Feed well, internal dart valve, launders, level and air control
300	PCA	20	M	Rougher Conc Pump A	Supplier: Metso Type: VF 350	92		froth factor 3.5
320	XCB	02	M	1st Cleaner 1 Cell 1	Supplier: Metso Type: RCS 20	37		c/w internal dart valve, launders, level and air control
330	MBB	02	M	Regrind Mill	Supplier: Metso Type:	1875		Tower mill
330	YAA	08	M	Regrind Cyclone	Supplier: Linatex Size: 6 x 375mm			5 operating, 1 s/by
330	PCB	18	M	Regrind Mill Discharge Pump	Supplier: Metso Type: HR250	90		
350	ACA	06	M	Concentrate Thickener	Supplier: Westpro Machinery Size: 20m dia	12		
400	TBA	04	P	CIL Tank #1	Supplier: DRAA Type:			625 m3 total volume
400	XSA	18	M	CIL Tank #1 Agitator	Supplier: Hayward Gordon Size: LH9	30		
420	TCB	14	M	Elution column	Supplier: Summit Valley Equipment Size:			
420	XWA	42	M	Electrowinning Cell	Supplier: Summit Valley Equipment Size:			
420	XGR	46	M	Gold Room (complete)	Supplier: Summit Valley Equipment Size:	125		
450	TBA	02	P	CN Destruction Tank #1	Supplier: DRAA Type:			40 m3 total volume
450	XSA	06	M	CND #1 Agitator	Supplier: Hayward Gordon Size:	18		
450	PPA	12	M	Leach Tailings Discharge Pump	Supplier: Metso Size: HR150	18		
500	TBA	14	P	Process Water Tank	Supplier: DRAA Type:			200 m3 total volume
500	PCC	16	M	Process Water Pump	Supplier: Metso Type: MM400	150		
500	HAC	32	M	Plant Air Compressor	Supplier: Ingersoll Rand Size: GA110	110		
500	HBB	50	M	Flotation Air Blower	Supplier: Continental Size: 600-5	110		no accoustic hood, but sound-reducing room
500	HBB	51	M	Cyanidation Air Blower	Supplier: Continental Size: 600-5	110		no accoustic hood, but sound-reducing room



EQUIPMENT LISTING

B	REGRIND MILL MODIFICATION	AS		ARH	JUN 2010
A	FOR CLIENT REVIEW	AS		ARH	JUN 2010
REV	REVISION DESCRIPTION	BY	CHKD	APPRV'D	DATE

SIGNATURE
DATE

PROFESSIONAL ENGINEER	CLIENT APPROVAL
NUMBER SIGNATURE DATE	SIGNATURE DATE

	NAME	DATE	CLIENT APPROVAL		
			DATE	NAME	SIGNATURE
ENG. MAN.					
PROJ. ENG.					
PROC. ENG.					
PROF.ENG.					
SECT. LDR.					
ELEC. ENG.					
MECH. ENG.					
INST.ENG.					

CAD REF No.– 10SPAN0100/DRAWINGS/10SPAN0100PFD100B.DWG

AGP Mining Consultants Inc.

SPANISH MOUNTAIN GOLD

FLOTATION / CARBON IN LEACH CIRCUIT
PROCESS FLOW DIAGRAM

	NAME	DATE	SCALE	SIZE	DWG. No.	REV.
DRAWN			NTS	11x17	10SPAN0100-PFD-100	B
CHECKED						

APPENDIX C

Geotechnical

BGC ENGINEERING INC.
AN APPLIED EARTH SCIENCES COMPANY

SPANISH MOUNTAIN GOLD LTD.

SPANISH MOUNTAIN PRELIMINARY ECONOMIC ASSESSMENT

ENGINEERING GEOLOGY AND GEOTECHNICAL EVALUATIONS

FINAL

PROJECT NO:	0697-005	DISTRIBUTION:	
DATE:	November 26, 2010	SPANISH MOUNTAIN GOLD	2 copies
DOCUMENT NO:	SM-10-01	BGC	2 copies
		AGP MINING	1 copy

TABLE OF CONTENTS

LIST OF TABLES

LIST OF DRAWINGS

LIST OF APPENDICES

LIMITATIONS OF REPORT

BGC Engineering Inc. (BGC) prepared this document for the account of Spanish Mountain Gold Inc. The material in it reflects the judgment of BGC staff in light of the information available to BGC at the time of document preparation. Any use which a third party makes of this document or any reliance on decisions to be based on it is the responsibility of such third parties. BGC accepts no responsibility for damages, if any, suffered by any third party as a result of decisions made or actions based on this document.

As a mutual protection to our client, the public, and ourselves, all documents and drawings are submitted for the confidential information of our client for a specific project. Authorization for any use and/or publication of this document or any data, statements, conclusions or abstracts from or regarding our documents and drawings, through any form of print or electronic media, including without limitation, posting or reproduction of same on any website, is reserved pending BGC's written approval. If this document is issued in an electronic format, an original paper copy is on file at BGC and that copy is the primary reference with precedence over any electronic copy of the document, or any extracts from our documents published by others.

1.0 INTRODUCTION

BGC Engineering Inc. (BGC) has been retained by AGP Mining Consultants Inc. (AGP), on behalf of Spanish Mountain Gold Ltd. (Spanish Mountain) to provide preliminary geotechnical interpretations to support the development of mining cost estimates for the Spanish Mountain Project near Likely, British Columbia. This report summarizes the preliminary engineering assessments conducted by BGC for the Spanish Mountain property.

1.1. Background and Scope of Work

At the request of AGP, BGC provided a proposed scope of geotechnical work, dated 30 April 2010. This was subsequently included in a proposal from AGP to Spanish Mountain dated 5 May, 2010. The scope of work included a site visit with representatives from AGP and Spanish Mountain, examination of drill core, and compilation of geotechnical and structural geologic data made available by Spanish Mountain. BGC has used this information to provide scoping level geotechnical assessments for a potential open pit at the Spanish Mountain property.

1.2. Previous Work by Others

Exploration drilling, reconnaissance mapping, and geochemical sampling were completed by Skygold Ventures Ltd. on Spanish Mountain from 2005 to present.

From 2005 to 2007, exploration drilling was focused on the North and Main Zones (Drawing 1). A total of 7,746 metres of diamond drilled was completed in 35 holes, and 3,377 metres of reverse circulation (RC) drilling was completed in 30 holes in 2005. This work delineated the basic deposit shape and gold mineralization distribution, and has been used as a basis for further exploration. In 2006, geological mapping, rock and soil sampling, exploration drilling, airborne geophysics and orthophotography were completed on a property-wide basis. A further 21,881 metres of diamond drilling was completed in 88 holes, and 5,009 metres of RC drilling was completed in 50 holes. In 2007, a total of 26,993 meters of diamond drilling was completed in 126 holes, focusing primarily on in-fill work within the Main Zone. Metallurgical tests were conducted on material acquired from this diamond drilling program.

In 2008, geological mapping, diamond drilling, and rock sampling and soil sampling programs were expanded to two new zones. A total of 40,449 metres of diamond drilling was completed in 161 holes within the Main Zone, the ROG Zone, and the CCR Zone. The southern portion of the Main Zone and the eastern edge of the ROG Zone were targeted for soil sampling where further work may define future drill targets.

In 2009 Spanish Mountain Gold personnel conducted geotechnical core logging under the guidance of Knight Piesold. A geotechnical data collection manual and core logging instruction sheet was provided to Spanish Mountain personnel during the 2009 drilling program.

1.3. Data Sources

The preliminary engineering geology interpretations and geotechnical studies presented in this report have been developed based on:

- Site geology maps and sections provided by Spanish Mountain included in a Technical Report on Resource Estimation on the Spanish Mountain Gold Deposit; NI 43-101 Technical Report- Skygold Ventures Ltd., 2009).
- Preliminary geotechnical data collected by Spanish Mountain during logging, including core recovery, RQD, hardness, fracture count, lithology and alteration data.
- A report entitled "Structural Interpretations" by Georgina Price, M.Sc. P. Geo., which was based on 2006 diamond drilling and surface mapping by Skygold Ventures Ltd., and trench mapping by Cyprus Canada Inc., Mt Calvery Resources Ltd., and Wildrose Resources Ltd., dated January, 2008.
- Core photographs taken by Spanish Mountain.
- Summary geotechnical logging of selected drill holes during a two day site visit by BGC.

The reliability of this data for geotechnical design varies; however, they are considered adequate for the purpose of developing preliminary engineering geology interpretations to support scoping level geotechnical evaluations for preliminary economic assessments.

1.4. Current Study and Limitations

The current report summarizes information and knowledge gathered to date, primarily by others, along with information collected by BGC during a two day site visit. This information provides the basis for preliminary pit slope design angles to assist in determining mining costs for the project.

A comprehensive drill hole and surface mapping database compiled by Spanish Mountain, and partially verified during field work completed by BGC, is the primary source of information. The accuracy of the dataset could not be verified during the site visit, particularly surface mapping information, and therefore the data is considered to be of moderate reliability until further field work can be conducted to confirm the measurements.

Where data gaps exist, the engineering geology of the area has been inferred from available data. When quantifying material properties of the rock, ranges of values have been estimated. Future assessments by mine planning personnel should consider incorporating sensitivity analyses, using the estimated ranges of key properties controlling the designs, i.e. rock mass strength and structural discontinuity orientations, to evaluate the potential impacts on economics due to uncertainties in those key properties.

Engineering geology interpretations presented in this report should be considered preliminary and, where appropriate, geological features identified should be verified and validated with additional field work and interpretation. Data used to provide initial quantitative estimates of the rock mass properties have primarily been collected within the Main and North zones and exploration drilling has concentrated on the mineralized horizon. This data may not accurately reflect the rock mass comprising the final open pit walls. The geomechanical properties of the rocks outside the ore zone could control the final excavation geometry and have a significant impact on mining economics. Recommendations for improving the quality of available data and reliability of the engineering geology interpretations for more detailed mine designs are provided later in the report.

2.0 2010 FIELD PROGRAM

Messrs. W Newcomen, P. Eng. and J Whittall of BGC visited the site from 11 May to 13 May 2010. During that period the following tasks were conducted:

- Rock mass characterization of select core intervals.
- Review local and regional structural geology.
- Collection of mapping and oriented core data.
- Review of rock and alteration types with Spanish Mountain geologists.

In total, 830 m of core from the Spanish Mountain deposit area were examined. Details regarding the drill hole identities and intervals of holes logged are included in Appendix A. The locations of the drill holes logged, and the surface outcrops mapped by Spanish Mountain are shown in Drawing 2.

3.0 SITE GEOLOGY

The Spanish Mountain property is a sediment hosted gold deposit in the Quesnel Terrane. It is composed of sedimentary and volcanic rocks from the Middle Triassic to Early Jurassic, specifically the Quesnel River Group and Takla Group. Both regional and local fold and fault structures are complex and poorly defined, however it is believed gold mineralization is stratigraphically and structurally controlled. Three persistent sub-parallel faults may have been conduits for mineralizing fluids. Disseminated gold appears to be concentrated in a 400 m wide chute around a central fault in the argillite, and in considerably narrower corridors in less permeable materials (siltstone and greywacke). Gold mineralization tends to follow north-westerly dipping (35˚-45˚) faults (Peatfield et al., 2009).

3.1. Lithology

For the purposes of this study the main lithologies of the deposits have been grouped into: siltstone, argillite, greywacke, and conglomerate. The distribution of these rock types in the North Zone and the Main Zone are shown in Drawings 3 and 4, and 5 and 6, respectively. It was initially assumed that rocks within these groupings would generally have the same geotechnical properties. However, preliminary investigations into the variation in structural control indicate that the hanging wall and footwall argillites may have different geotechnical properties. This is discussed briefly below and may warrant additional investigations in the next phase of work.

The specific rock types located in the deposit area are summarized below. Pertinent characteristics (i.e. mineralogy, alteration, etc.) of the various lithologies, as they pertain to their geotechnical properties, are outlined in the following sections.

The geologic units encountered in the Spanish Mountain deposit area are described in the following paragraphs.

Siltstone - This unit is finely laminated with frequent mafic dykes, decreasing with depth. It is medium to light grey with sericite alteration and local strong quartz veining. The siltstone makes up a large component of the rocks anticipated in the North Zone (Drawings 3 and 4).

Argillite – This unit is fine, black to dark grey, with siltstone interbeds and locally finely laminated. Iron-magnesium carbonate alteration is present within this unit at varying intensities. Argillite hosts much of the disseminated gold mineralization in the Main and North zones. The south wall of the Main Zone Pit is anticipated to be primarily in the footwall argillite (Drawing 5).

Greywacke – This unit is a fine to coarse, light gray, sequence of wacke. Fine laminations are observed but largely concealed by intense serictic and iron-magnesium carbonate alteration. A relatively large component of the east and west walls of the Main Zone Pit are anticipated to be in the greywacke (Drawing 6).

Conglomerate – This unit is a granule conglomerate which becomes finer grained with depth. Clasts are mainly siltstone and greywacke in a very fine matrix. Generally this unit can be used as a marker bed for the greywacke-footwall argillite contact.

3.2. Alteration

Hydrothermal alteration associated with gold-copper porphyry deposits is common. Alteration facies observed at Spanish Mountain include:

- Graphitic
- Carbonate
- Argillic
- Phyllic
- Ankeritic
- Sericitic
- Siliceous

To date the relationship between various alteration types and the geotechnical properties of the rock types have not been determined for the Spanish Mountain property.

4.0 GEOTECHNICAL ASSESSMENTS

4.1. Geotechnical Database

A geotechnical database has been compiled which includes data from work completed by Spanish Mountain, Knight Piesold, regional geology mapping, and more recent work undertaken by BGC. This database has been used to estimate geomechanical design parameters for the proposed open pit.

Based on the available data, the rock mass has been divided into geotechnical units for design purposes. At this preliminary stage of the project the geotechnical units have been grouped according to lithology (i.e. rock type). Preliminary indications suggest that alteration may play a role in the geotechnical characteristics of the greywacke and siltstone; however, further evaluations are required before alteration type can be used to assist in open pit design.

4.1.1. Core Logging

Spanish Mountain completed five hundred and eighty six (586) exploration core holes in the vicinity of the deposit. A total of 68,195 m of core has been drilled to date. Rock Quality Designation (RQD) was logged by Spanish Mountain for holes drilled since 2007, with fracture count and hardness added to the core logging information collected in 2009. BGC logged additional geotechnical parameters on selected intervals of split (sawn) core during the site visit. The locations of the holes which BGC partially logged are shown on Drawing 1. Geomechanical information collected from the drill core by BGC is summarized in Table 1. Cumulative frequency plots of RQD and Fracture Intercept data collected by Spanish Mountain are also included in Appendix A showing the distribution of these parameters by primary rock type.

4.1.2. Outcrop Mapping

Surface mapping data with outcrop coordinates was provided by Spanish Mountain during the site visit. This data included lithology, structure type, structure orientation, and location. The locations of the structural geologic mapping measurements are shown in Drawing 1. No additional surface mapping was carried out by BGC for this study.

4.1.3. Intact Rock Characterization

Properties of the intact rock for each of the main rock types were estimated from core observed during the field visit and from geotechnical core logging data collected by Spanish Mountain. Unconfined compressive strengths (UCS) for the various rock types in the resource area were estimated in the field using standard index tests (ISRM, 1978) and are summarized in Table 1, along with a other pertinent rock mass rating parameters collected by Spanish Mountain and BGC.

4.2. Rock Mass Strength

To determine design properties for each geotechnical unit, the intact rock characteristics for each lithology are scaled according to the density of the discontinuities and the character of the rock mass fabric. The scaling is accomplished using the Geological Strength Index (GSI) determined from outcrop mapping (Marinos, et. al., 2005), or a rock mass rating (RMR) determined from geotechnical core logging (Bieniawski, 1976). The design shear strength and deformation modulus of each geotechnical unit have been estimated according to well established empirical methods (Hoek et al., 2002), which utilize GSI and RMR, and the following additional input parameters:

- Uniaxial or unconfined compressive strength (UCS) of the intact rock.
- Material constant (m_i) of the intact rock (which can be estimated from either published values or laboratory testing, if it has been conducted).
- Estimate of disturbance ("D") of the rock mass due to excavation.

UCS values for the lithologic units have been previously discussed. The Hoek-Brown material constant (mi) reflects the indurations, grain or crystal interlocking, and mineralogy of the intact rock sample. Values of m_i for each rock type have been assumed based on published values (Hoek, 2007), and engineering judgement (Table 2), as follows:

- m_i for the conglomerate = 21.
- m_i for the greywacke rocks = 18.
- m_i for the argillites = 7.
- m_i for the siltstones = 7.

A disturbance factor ("D") has been applied to the rock mass to represent the effects of mining. Blast damage, stress relief, and mining induced relaxation will affect the structural fabric of the geotechnical units, causing dilation of geologic structures and occasionally inducing additional fracturing of the rock mass. The disturbance factor has been assumed to be 0.85, equivalent to the disturbance resulting from mining using traditional drill and blast methods.

4.3. Structural Geologic Model

A relatively limited amount of structural data has been collected from regional geologic mapping, outcrop mapping and oriented core drilling conducted by Spanish Mountain. The locations of structural measurements collected by Spanish Mountain, from both surface exposures and from oriented core measurements are presented on Drawing 1.

Data provided to BGC during the site visit has been plotted on equal area, lower hemisphere stereonets for each of the deposits (Appendix B) using the commercially available software DIPS (RocScience, 2008). Structural discontinuity data from the Main and North zones have been separated.

The quantity of structural data is limited, particularly for fractures and faults; however, sufficient data is available to make some preliminary interpretations regarding the structural fabric of the rocks which will be encountered in the proposed open pit. Bedding measurements are relatively abundant and have been assumed to be more reliable and predictable due to regional mapping efforts undertaken in the area.

Based on the structural data presented in the stereonets (Appendix B), the following can be stated:

- Bedding orientations exhibit a substantial (20°-50°) rotation between the Main and North Zones, suggesting a structural domain boundary may be present between the two zones.
- Bedding in the Main Zone (Appendix B) dips primarily towards the southwest (194°) at an average angle of about 30°. A second weaker concentration of bedding planes is oriented almost due north at an angle of about 40˚. The bedding discontinuities could be a primary structural control on interramp slopes in the north wall of the proposed Main Zone pit, if the bedding is continuous.
- Bedding in the North Zone is highly variable, with a wide range of dip directions from the northeast to the southwest. Three main sets have been identified with average dip angles ranging from 32˚ to 43˚.
- Based on information obtained from the structural geologic mapping and oriented core, south-southwest and north-northeast facing pit walls could be subject to structurally controlled instability due to the strong presence of bedding.
- In general the core orientation data appears to be supported by the surface mapping data although there is evidence of significant folding and variation of bedding with depth. This is supported by our observations from shallow test pits in rock in the project area.

5.0 GENERAL COMMENTS ON ROCK MASS QUALITY AND STRENGTH

Based on the core logging and mapping information provided by Spanish Mountain and our inspection of the core the following general comments apply:

- Based on the core logging data provided, there appears to be little difference in the rock mass quality of the siltstone, argillite and the greywackes. However, our observations during the site visit suggest that the greywacke and conglomerate units are of slightly better quality than the other rocks in both the Main and North zones.
- The most distinct difference in rock mass strength is observed in the fault zones, which are very weak.
- The orientation of several wide graphitic altered fault zones with significantly lower rock quality is poorly defined and will likely be a control on interramp slopes.
- Alteration types may also play a role in the strength of the rocks, with argillic, graphitic, carbonitic and phyllic alterations generally resulting in a lower quality rock mass than rocks with siliceous, sericitic and ankeritic alterations. The relationship between alteration type and rock quality indicates that it may be worthwhile to further evaluate the role of alteration in pit slope design angles at the next stage of study.

It is noteworthy that the current geologic logging system appears to be relatively insensitive to some of the geotechnical parameters used for slope designs.

6.0 PRELIMINARY OPEN PIT DESIGN CRITERIA

6.1. General

There are two main controls on achievable open pit slope design angles. The first consideration is the potential for structural instabilities, whereby discontinuities in the rock mass (joints, bedding planes, faults, etc.) intersect the excavation such that it becomes "kinematically possible" for failure to occur (i.e. the geologic discontinuities daylight out of the slope). Achievable slope angles are therefore limited by the orientation and the shear strength of the discontinuities. Structurally controlled slope failures can occur at any scale, i.e. at the bench, interramp, and the overall slope scales.

The second consideration is the strength of the rock mass. This is dictated by the amount of fracturing within the rock mass, the characteristics of the discontinuities, and the intact rock strength. Rock mass stability generally includes large-scale, deep-seated failures and slope-scale failures through weak geological units.

6.2. Structurally Controlled Instability

The structural geologic model of the Spanish Mountain deposit and surrounding area is still being developed, and relatively limited information was available at the time the PEA was undertaken. As a result, only a cursory evaluation of the impacts of geologic structure on pit wall stability could be undertaken.

As discussed in Section 4.4, structural discontinuities at the interramp scale could control achievable interramp angles on north facing slopes in the Main Zone, and for northeast to southwest facing slopes in the North Zone. At this preliminary stage of design, it is recommended that bedding should not be undercut where the average dip is greater than 30°, in order to minimize the potential for structurally controlled instability. This applies to Design Sector MZ-180 (slope azimuth 150° to 210°) in the Main Zone and Design Sectors NZ-238, NZ-315 and NZ-025 (slope azimuth 190° to 065°) in the North Zone. Lower hemisphere equal area stereonets showing the structural discontinuity populations and the preliminary design sectors are included in Appendix B.

Despite considerable scatter in the bedding orientations, both the oriented core and surface mapping data in the Main Zone and North Zones identify two to three prominent bedding orientations. Depending on the local continuity of the bedding and the shear strength of these discontinuities, bench scale failures may occur for these pit wall orientations, and occasional wide berms may be required to contain the failures. However, until additional information on the continuity and spacing of the bedding is available the confidence level of the bench designs, i.e. their ability to contain bench-scale instability, will be low. The variability of the bedding orientations needs to be further evaluated to determine whether or not this has an overall positive or negative effect on the achievable interramp and overall slope angles.

6.3. Rock Mass Instability

Generic stability analyses were carried out to evaluate various pit wall geometries, with slope heights ranging from 100 m to 500 m, and overall pit wall angles ranging from 30° to 60°. The rock mass strength parameters, assigned to the three primary geological units (siltstone, argillite and greywacke), are summarized in Table 1. For the generic stability analyses the rocks comprising the pit walls were assumed to be homogeneous with no structural controls, i.e. the site specific geology was not incorporated into the cross-sections, to simplify the stability analyses. The results of the generic stability analyses can be used to provide broad guidance to mine planners on achievable overall slope angles within the primary rock types. The design curves (Drawing 7) can also be used to provide guidance on interim pit wall angles, should mine planning considerations require that they differ from the ultimate pit wall angles.

Hydrogeologic conditions for the Spanish Mountain deposit are not well defined. However, at this preliminary economic assessment stage it has been assumed that the slopes have been completely dewatered. Therefore, the generic, rock mass stability analyses have been conducted assuming dry conditions, i.e. with a pore pressure coefficient (Ru) of zero. Relatively low interramp heights of 200 m and 100 m have been assumed for the Main and North Zones, respectively, to facilitate effective depressurization of the pit walls by allowing relatively frequent dewatering well installation as the pit is deepened.

Factors of safety (FOS) were calculated for various slope heights and angles using the limit-equilibrium method of slices in SLIDE (RocScience, 2008). The limit equilibrium method does not take into account the effect of in-situ stress on the overall slope stability. The limit equilibrium method also does not account for material strain and displacement and thus the modeled factor of safety only considers the average shear stress mobilized along the slope due to the rock mass strength properties. Slope height/slope angle combinations resulting in a FOS between 1.2 and 1.3 have been plotted to show the acceptable slope angle for a given slope height under both dry and partially saturated conditions (Drawing 7).

Based on the estimated rock mass strength of the argillites in the Main Zone, overall pit wall angles of 32° to 43° are predicted to be feasible for pit wall heights between 250 m and 500 m (Drawing 7). Based on the estimated rock mass strength of the greywacke, significantly steeper overall slopes could be achieved; however, it appears that the critical south wall will be primarily in the footwall argillites (Drawing 5). Note that the predicted achievable overall slope angles assume that there is no structural control on the potential failures. Unfavourably oriented geologic structures are likely present locally, and additional structural geologic investigations/interpretations are necessary to gain greater confidence in these recommended wall angles.

Based on the estimated rock mass strength of the siltstone in the North Zone, and assuming “dry” conditions, overall pit wall angles ranging from 42° to 55° could potentially be achieved for slope heights from 100 m to 200 m. However, partially saturated conditions are likely more reasonably assumed due to the presence of Spanish Creek nearby, in which case considerably shallower pit wall angles of between 37° and 48° are predicted. Regardless of the assumptions with respect to groundwater pressures in the North Zone, a high degree of depressurization will be required to achieve reasonable slope angles in the siltstone. Groundwater pressures will need to be more accurately quantified in the proposed pit walls before greater confidence can be gained in the design angles for these materials.

7.0 SUMMARY AND CONCLUSIONS

The preliminary engineering geology of the Spanish Mountain deposit has been summarized to provide a basis for scoping level mine planning and preliminary economic assessments. BGC has developed a basic description of the expected geologic materials of the resource area from available maps, geologic descriptions by Spanish Mountain, core hole data, and field review.

The five preliminary geotechnical units for mine design are: siltstone, argillite, greywacke, conglomerate, and fault zones. Relatively limited data is available regarding the rock mass strength and the geologic structure in the Spanish Mountain deposit. The main limitations to the data are as follows:

- All of the core available for inspection has already been cut in half. In order to conduct accurate geotechnical investigations the core should be intact.
- In general, the joint spacing and rock strength for all units are not well defined, as previous logging techniques were insensitive to these properties.
- Structural geologic information is relatively sparse.

Sufficient data has been compiled regarding geotechnical strengths of the primary rock types to provide a range of potential pit wall angles for use in the preliminary economic assessment. However, in order to develop the slope design angles presented in this report, numerous assumptions had to be made about the potential primary controls on slope stability, the geology, the strength of the rock mass, the groundwater pressures and the potential failure mechanism. The following assumptions were made:

- Interramp slope angles could be limited due to structurally controlled failures along continous bedding.
- Anisotropy of the rock mass was not considered in the generic (i.e. rock mass) stability analyses conducted.
- Groundwater pressures were assumed to be a function of the lithostatic stress.

Structural controls at the interramp scale will likely exist for southwest dipping slopes. In order to avoid undercutting bedding, it is recommended that interramp slope angles be limited to 30° in the northeast walls of potential open pits. However, overall slope heights greater than about 200 m in the siltstone could be limited by the rock mass strength to angles as low as 35° unless aggressive depressurization of the siltstone is implemented, should pore pressures exist in these rocks. However, the potentially achievable overall slope angles assume that there is no structural control on the potential failures.

8.0 RECOMMENDED FUTURE WORK

The data available at this stage of study varies in reliability. Estimates of engineering properties are provided with ranges where possible and sensitivity analyses are encouraged for geotechnical design based on these data. The interpretations of this report are preliminary and require addition validation and testing with higher quality data before they can be applied to higher level design studies.

For pre-feasibility design studies, greater confidence in the geotechnical input parameters will be required and the preliminary geotechnical model presented will need to be updated with additional data. A series of recommended data collection and interpretation tasks are outlined below. These recommendations could be completed in phases using a combination of BGC/AGP staff, Spanish Mountain staff, and/or other contractors.

8.1. Outcrop Mapping

Mapping of additional exposed rock outcrop along drill roads or other access roads could provide important data on discontinuity orientation, character, and continuity; which are all critical for rock excavation design. Further information on the quality of the rock mass and the character and thickness of the overburden and/or oxidized rocks should be collected.

8.2. Geotechnical Core Logging

Six to seven dedicated geotechnical core holes were proposed to target the proposed PEA level mining excavation and have already been drilled. These holes have mainly targeted waste rock outside of the ore zone to determine the geotechnical properties of the rock mass forming the pit walls. The core holes were drilled using a triple tube core barrel. Core orientation techniques or a televiewer system were employed to determine sub-surface geologic discontinuity information for comparison with surface mapping information.

The existing core hole database does not provide sufficient geotechnical data to characterize the rock mass of the resource area according to standard rock mechanics techniques. Thus, BGC has implemented a core logging procedures be modified to provide complete parameters for RMR (Bieniawski, 1976). We recommend that the joint roughness characteristic (JRC), as defined by Barton and Choubey (1977) be collected on the discontinuities logged as part of the core orientation work. The JRC can be used to estimate the friction angle of the discontinuity which will assist in kinematic stability analyses for slope designs.

8.3. Point Load Testing

Experiences at other mining properties and published literature have indicated that alteration may have a significant impact on the intact strength of the rocks of the resource area. The potential for further division of the geotechnical units according to alteration may be evaluated through a point load testing program. The point load test is a simple and rapid method of determining an index value which can be related to the intact strength of a rock sample. This index may be used to relatively compare intact strength variation according to alteration.

Spanish Mountain could undertake a point load testing program as part of its next exploration drilling program. The added effort is minimal, requiring the rental or purchase of a point load test machine, the selection of samples according to lithology and alteration type, and completing the testing itself. BGC can provide further support in developing the testing specifications and database to store the results.

Discrepancies between estimated of UCS from field hardness grade, point load testing and laboratory testing (see below) will need to be resolved in the next phase of work.

8.4. Laboratory Testing

Uniaxial compressive strength testing, direct shear testing of discontinuities, Brazilian tensile strength testing, and index testing of discontinuity infill should be conducted on select samples to provide a basis for geotechnical analysis and design parameters. These samples should be collected from dedicated geotechnical drill holes to ensure the appropriate materials are sampled, and to avoid conflicts with exploration sampling and assaying.

8.5. Hydrogeologic Evaluations

Hydrogeological testing (packer testing) and instrumentation (i.e. piezometers) should be installed in select holes to provide basic data for groundwater modeling and excavation dewatering / depressurization simulations. This information will be useful in subsequent geotechnical evaluations, to determine the feasibility of dewatering the proposed pit.

8.6. Costs

Depending on the actual drilling time required to complete the geotechnical drilling, anticipated costs for pre-feasibility level investigations could range from about $500,000 to $600,000. Disbursements for the field and laboratory testing components of the work typically range from $75,000 to $125,000 and are included in this total.

9.0 CLOSURE

We trust the above satisfies your requirements at this time. Should you have any questions or comments, please do not hesitate to contact us.

Yours sincerely,

BGC ENGINEERING INC.
per:

ISSUED AS DIGITAL DOCUMENT.
SIGNED HARDCOPY ON FILE WITH
BGC ENGINEERING INC.

Warren Newcomen, M.S., P.Eng.
Senior Geotechnical Engineer

John Whittall, E.I.T
Junior Geotechnical Engineer

Reviewed by:

Roland J. Tosney, P.Eng.
Senior Geotechnical Engineer

REFERENCES

Barton, N.R., Lien, R. and Lunde, J. 1974. Engineering classification of rock masses for the design of tunnel support. Rock Mechanics 6(4): 189-239

Bieniawski, Z.T. 1976. Rock mass classification in rock engineering. In Exploration for rock engineering, proc. of the symp., (ed. Z.T. Bieniawski) 1, 97-106. Cape Town: Balkema

Hoek, E., Carranza-Torres, C., and Corkum, B., 2002. Hoek-Brown failure criterion – 2002 Edition, In: Proceedings of the NARMS-TAC Conference, Toronto, pp. 267-273.

Hoek, E., 2007. Chapter 11 – Rock Mass Properties. In: Practical Rock Engineering.

International Society for Rock Mechanics (ISRM). 1978. Suggested methods for the quantitative description of discontinuities in rock masses. International Journal of Rock Mechanics, Mining Sciences & Geomechanics Abstracts, 15: 319 – 368

Rocscience, 2008. RocLab v. 5.1. Rocscience Inc. Toronto, Canada.

Rocscience, 2008. Slide v. 5.0 Rocscience Inc. Toronto, Canada.

TABLES

TABLE 1. ROCK MASS RATING SUMMARY

Geotechnical Unit	RQD (%)[1]	RQD Rating[4]	Fracture Intercept (m)[1]	Fracture Intercept Rating[4]	Strength Grade[2]	Strength Grade Rating[4]	Joint Condition[3]	RMR '76[4]	GSI[5]
Greywacke	61	12.1	0.08	12.9	3.0	4	18	57.0	57.0
HW. Argillite	58	11.4	0.08	13.0	3.0	4	16	54.4	54.4
FW. Argillite	54	10.8	0.08	13.1	3.0	4	18	55.9	55.9
Siltstone	38	8.0	0.07	12.4	3.0	4	16	50.4	50.4
Conglomerate	65	12.7	0.07	12.7	3.0	4	20	59.4	59.4
Fault Zones	8	3.9	0.06	12.1	1.0	1	6	30.0	30.0

Notes:

1. Rock mass rating input values estimated from geotechnical database provided by Spanish Mountain Gold.
2. Strength grade (hardness) as per ISRM (1978) estimated by BGC during the site visit.
3. Joint Condition as defined by Bieniawski, 1976.
4. RQD, Fracture Intercept, and Strength Grade Ratings are as per Bieniawski, 1976.
5. GSI = Geologic Strength Index, as per Marinus et al., 2000

TABLE 2. HOEK-BROWN PARAMETERS

Unit	GSI	Strength Grade	UCS (MPa)	Hoek-Brown Material Constant, m_i	mb	s	Unit Weight (KN/m^3)
Conglomerate	59	3.0	25.0	21	1.645	0.0017	26
Greywacke	57	3.0	25.0	18	1.246	0.0013	26
Argillite	54	3.0	25.0	7	0.402	0.0008	26
Siltstone	50	3.0	25.0	7	0.314	0.0004	26

Notes:

1. The Hoek-Brown Criterion have been estimated using a disturbance factor ('D') of 0.85 for all units.
2. Mean RMR'76 parameters are used for each unit.

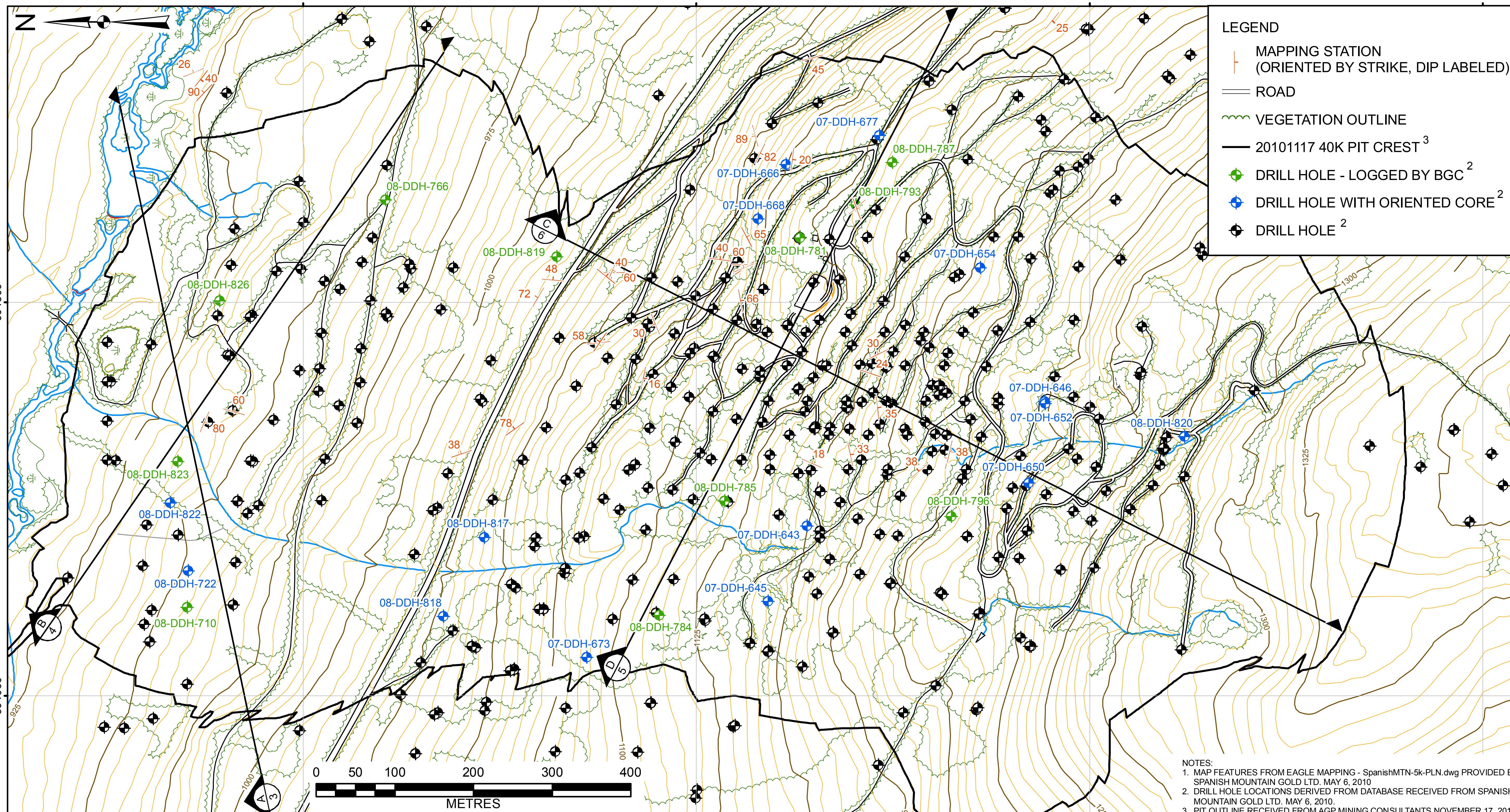
TABLE 3. SUMMARY OF RECOMMENDED OPEN PIT SLOPE DESIGN CRITERIA

Domain	Design Sector	Slope Azimuth		Maximum Interramp Height [1]	Bench Height[2]	Bench Face Angle[3]	Bench Width	Interramp Angle				Approximate Overall Slope Height[7]
								Bench Geometry[4] (G)	Kinematic[5] (K)	Rock Mass[6] (R)	Design Value	
		Start (°)	End (°)	Ih (m)	Bh (m)	Ba (°)	Bw (m)	Ia (°)	Ia (°)	Ia (°)	Ia (°)	Oh (m)
Main Zone	MZ-010	315	065	200	20	65	9.5	47	-	47	47	125
	MZ-108	065	150	200	20	65	9.5	47	-	47	47	225
	MZ-180	150	210	200	20	65	17.5	37	39	47	37	425
	MZ-263	210	315	200	20	65	9.5	47	-	47	47	300
North Zone	NZ-025	345	065	100	10	65	9.0	36	43	53	36	125
	NZ 128	065	190	100	10	65	9.0	36	-	53	36	225
	NZ-238	190	285	100	10	65	9.0	36	39	53	36	225
	NZ-315	285	345	100	10	65	12.0	31	32	53	31	125

Notes:

1. Maximum interramp height assumed based on typical pit dewatering and geotechnical instrumentation requirements. Lower interramp heights of 100m are required in North Zone due to the proximity to Spanish Creek.
2. Bench height provided by AGP Mining.
3. Bench face angle assumed based on average angle from BGC database of bench geometries.
4. Geometric control based on bench height = 20 m in Main Zone and 10 m in North Zone. Bench face angle = 65 ^{o} and bench width as shown.
5. Interramp slope angles limited by kinematic controls are based on the maximum angles that can be obtained without undercutting bedding, where bedding dips greater than 30^{o}. Bedding design sets are indicated in stereonets for the Main Zone and the North Zone in
6. Maximum allowable interramp angle due to rock mass quality based on assumed maximum interramp height in typical rock type for that domain. Interramp and maximum slope angles are based on assumed fully depressurized slopes (Ru=0).
7. Height estimated from pit plans provided by AGP Mining Consultants, November 17, 2010. Design curves presented in Drawing 7 should be used to determine maximum overall slope based on the overall slope heights, ensuring that the maximum interramp heights are not exceeded.

DRAWINGS



REV.	DATE	REVISION NOTES	DRAWN	CHECK	APPR.

SCALE:	1:5,000
DATE:	NOV 2010
DRAWN:	WKL,MIB
DESIGNED:	JW
CHECKED:	HWN
APPROVED:	HWN

PROFESSIONAL SEAL:

BGC ENGINEERING INC.
AN APPLIED EARTH SCIENCES COMPANY

CLIENT: SPANISH MOUNTAIN GOLD LTD.

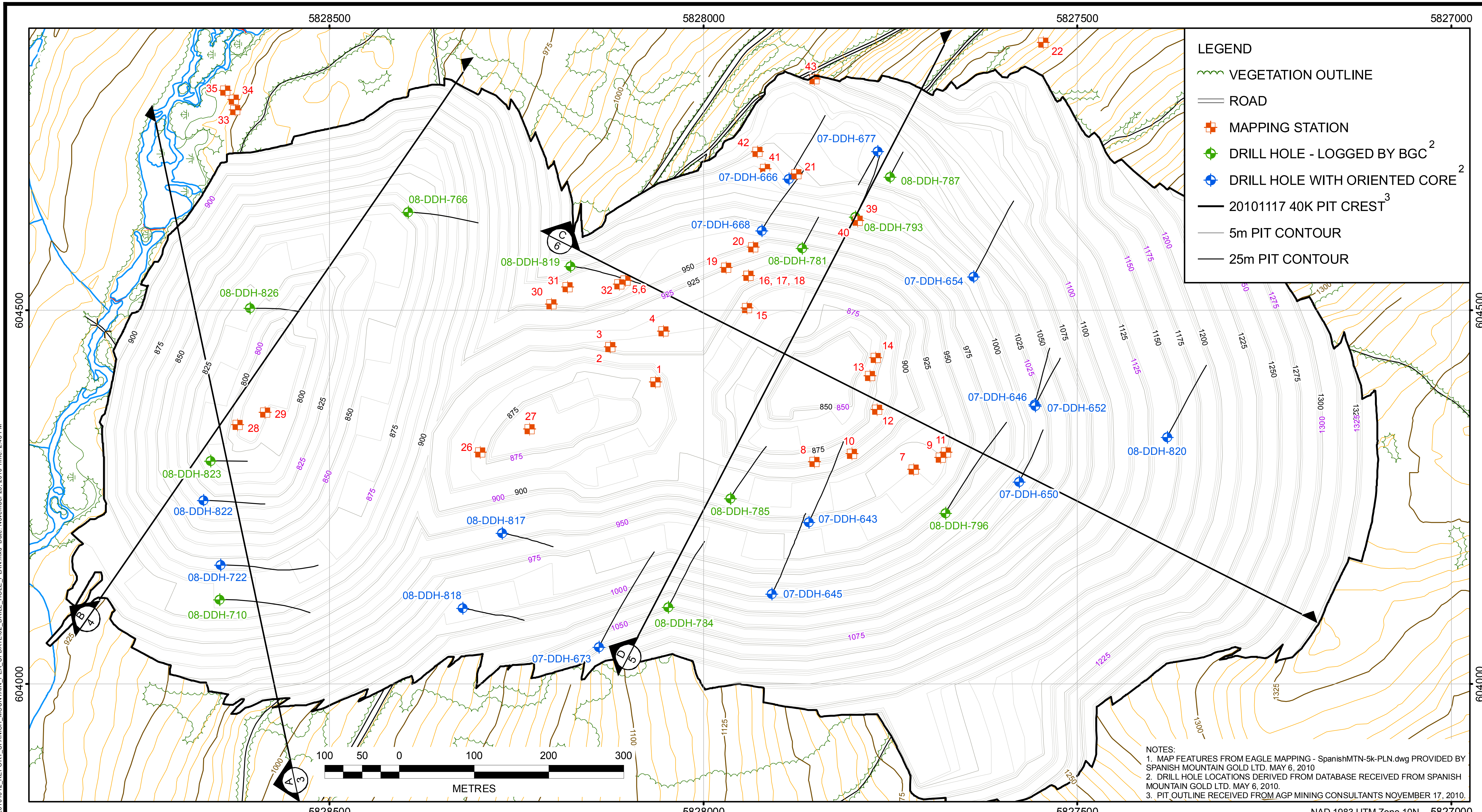
PROJECT: SPANISH MOUNTAIN PEA OPEN PIT SLOPE DESIGN

TITLE: DATA SOURCES FOR GEOTECHNICAL EVALUATIONS

PROJECT No.:	DWG No.:	REV.:
0697-005	1	A

X:\Projects\0697\005_Spanish_MtnWorkspace\20101012_REPORT_SPANISH_MOUNTAIN_PEA_UPDATE\01_MAPPING_STATIONS.mxd Date: November 25, 2010 Time: 2:46 PM

DWG TO BE READ WITH BGC REPORT TITLED "SPANISH MOUNTAIN PEA OPEN PIT SLOPE DESIGN" DATED NOVEMBER 26, 2010



REV.	DATE	REVISION NOTES	DRAWN	CHECK	APPR.

SCALE:	1:5,000
DATE:	NOV 2010
DRAWN:	WKL, MIB
DESIGNED:	JW
CHECKED:	HWN
APPROVED:	HWN

PROFESSIONAL SEAL:

BGC ENGINEERING INC.
AN APPLIED EARTH SCIENCES COMPANY

CLIENT: SPANISH MOUNTAIN GOLD LTD.

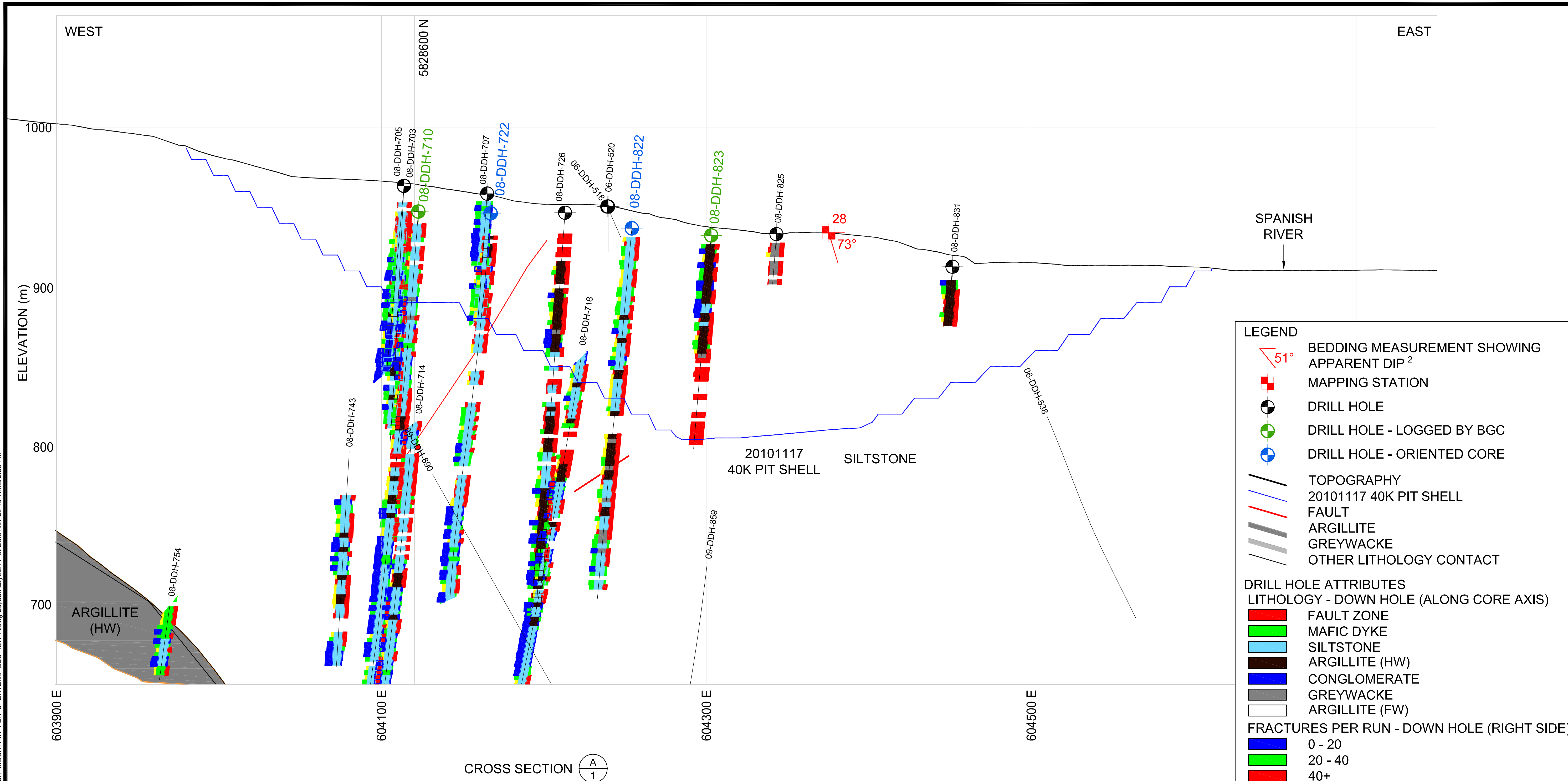
PROJECT: SPANISH MOUNTAIN PEA OPEN PIT SLOPE DESIGN

TITLE: DRILL HOLE PLAN AND PRELIMINARY PIT

PROJECT No.:	DWG No.:	REV.:
0697-005	2	A

DWG TO BE READ WITH BGC REPORT TITLED "SPANISH MOUNTAIN PEA OPEN PIT SLOPE DESIGN" DATED NOVEMBER 26, 2010.

X:\Projects\0697\005_Spanish_MtnWorkspace\20101012_REPORT_SPANISH_MOUNTAIN_PEA_UPDATE\02_DRILL_HOLE_PLAN.mxd Date: November 25, 2010 Time: 2:48 PM



Scale 1:2500

50 0 50 100 150

metres

NOTES:

1. PIT OUTLINE PROVIDED BY AGP MINING CONSULTANTS NOVEMBER 17, 2010.
2. CORE ORIENTATION INFORMATION COLLECTED BY SPANISH MOUNTAIN GOLD.
3. FRACTURES PER RUN AND RQD SHOWN ON DRILL HOLE BASED ON LOGGING CONDUCTED BY SPANISH MOUNTAIN GOLD.

REV.	DATE	REVISION NOTES	DRAWN	CHECK	APPR.

SCALE:	1:2500
DATE:	NOV 2010
DRAWN:	WKL
DESIGNED:	MK
CHECKED:	HWN
APPROVED:	HWN

PROFESSIONAL SEAL:

BGC ENGINEERING INC.
AN APPLIED EARTH SCIENCES COMPANY

CLIENT: SPANISH MOUNTAIN GOLD LTD.

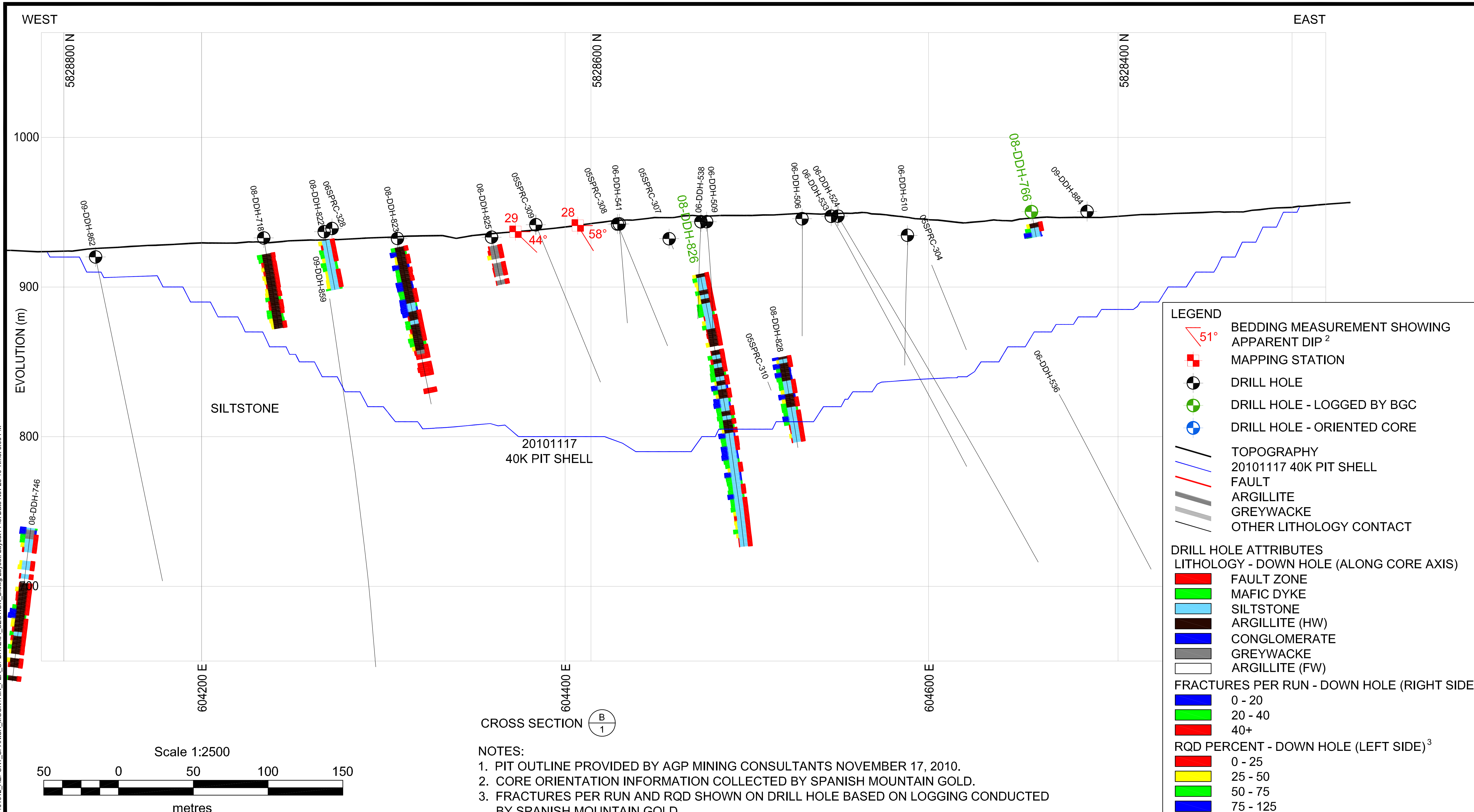
PROJECT: SPANISH MOUNTAIN PEA OPEN PIT SLOPE DESIGN

TITLE: GEOTECHNICAL SECTION A - NORTH ZONE

PROJECT No.:	DWG No.:	REV.:
0697-005	3	A

DWG TO BE READ WITH BGC REPORT TITLED "SPANISH MOUNTAIN PEA OPEN PIT SLOPE DESIGN" DATED NOVEMBER 26, 2010

X:\Projects\0697\005_Spanish_Mtn\Workspace\20101012_REPORT_SPANISH_MOUNTAIN_PEA_UPDATE\03_SECTION_A.dwg Layout: Layout1 Plot Date: Nov 25 10 Time: 2:53 PM



LEGEND

- BEDDING MEASUREMENT SHOWING APPARENT DIP [2]
- MAPPING STATION
- DRILL HOLE
- DRILL HOLE - LOGGED BY BGC
- DRILL HOLE - ORIENTED CORE
- TOPOGRAPHY
- 20101117 40K PIT SHELL
- FAULT
- ARGILLITE
- GREYWACKE
- OTHER LITHOLOGY CONTACT

DRILL HOLE ATTRIBUTES

LITHOLOGY - DOWN HOLE (ALONG CORE AXIS)
- FAULT ZONE
- MAFIC DYKE
- SILTSTONE
- ARGILLITE (HW)
- CONGLOMERATE
- GREYWACKE
- ARGILLITE (FW)

FRACTURES PER RUN - DOWN HOLE (RIGHT SIDE) [3]
- 0 - 20
- 20 - 40
- 40+

RQD PERCENT - DOWN HOLE (LEFT SIDE) [3]
- 0 - 25
- 25 - 50
- 50 - 75
- 75 - 125

NOTES:
1. PIT OUTLINE PROVIDED BY AGP MINING CONSULTANTS NOVEMBER 17, 2010.
2. CORE ORIENTATION INFORMATION COLLECTED BY SPANISH MOUNTAIN GOLD.
3. FRACTURES PER RUN AND RQD SHOWN ON DRILL HOLE BASED ON LOGGING CONDUCTED BY SPANISH MOUNTAIN GOLD.

SCALE:	1:2500
DATE:	NOV 2010
DRAWN:	WKL
DESIGNED:	MK
CHECKED:	HWN
APPROVED:	HWN

REV.	DATE	REVISION NOTES	DRAWN	CHECK	APPR.

PROFESSIONAL SEAL:

BGC ENGINEERING INC.
AN APPLIED EARTH SCIENCES COMPANY

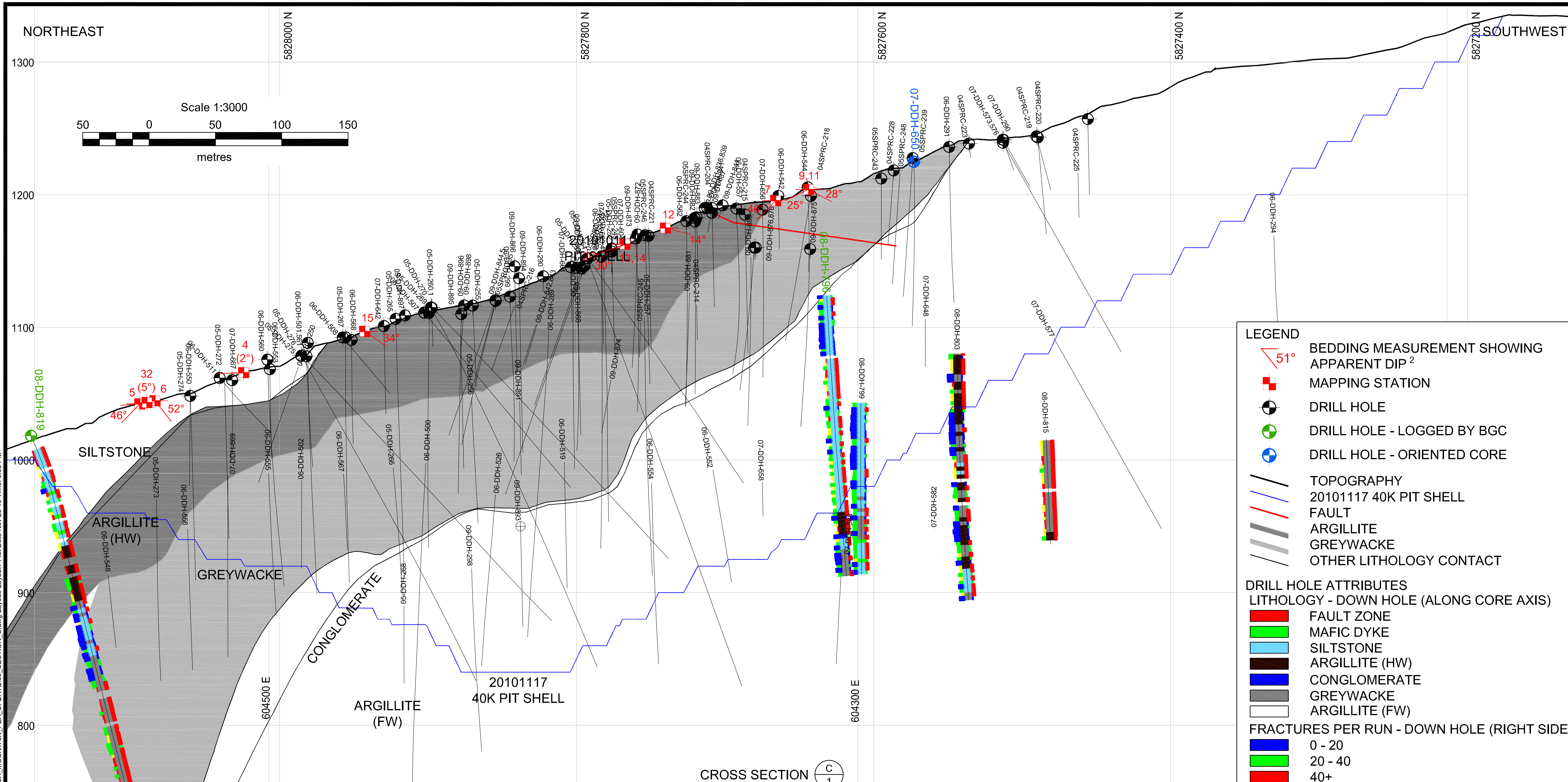
CLIENT: SPANISH MOUNTAIN GOLD LTD.

PROJECT: SPANISH MOUNTAIN PEA OPEN PIT SLOPE DESIGN

TITLE: GEOTECHNICAL SECTION B - NORTH ZONE

PROJECT No.:	DWG No.:	REV.:
0697-005	4	A

DWG TO BE READ WITH BGC REPORT TITLED "SPANISH MOUNTAIN PEA OPEN PIT SLOPE DESIGN" DATED NOVEMBER 26, 2010



LEGEND

- BEDDING MEASUREMENT SHOWING APPARENT DIP [2]
- MAPPING STATION
- DRILL HOLE
- DRILL HOLE - LOGGED BY BGC
- DRILL HOLE - ORIENTED CORE
- TOPOGRAPHY
- 20101117 40K PIT SHELL
- FAULT
- ARGILLITE
- GREYWACKE
- OTHER LITHOLOGY CONTACT

DRILL HOLE ATTRIBUTES

LITHOLOGY - DOWN HOLE (ALONG CORE AXIS)
- FAULT ZONE
- MAFIC DYKE
- SILTSTONE
- ARGILLITE (HW)
- CONGLOMERATE
- GREYWACKE
- ARGILLITE (FW)

FRACTURES PER RUN - DOWN HOLE (RIGHT SIDE) [3]
- 0 - 20
- 20 - 40
- 40+

RQD PERCENT - DOWN HOLE (LEFT SIDE) [3]
- 0 - 25
- 25 - 50
- 50 - 75
- 75 - 125

NOTES:
1. PIT OUTLINE PROVIDED BY AGP MINING CONSULTANTS NOVEMBER 17, 2010.
2. CORE ORIENTATION INFORMATION COLLECTED BY SPANISH MOUNTAIN GOLD.
3. FRACTURES PER RUN AND RQD SHOWN ON DRILL HOLE BASED ON LOGGING CONDUCTED BY SPANISH MOUNTAIN GOLD.

REV.	DATE	REVISION NOTES	DRAWN	CHECK	APPR.

SCALE:	1:3000
DATE:	NOV 2010
DRAWN:	WKL
DESIGNED:	MK
CHECKED:	HWN
APPROVED:	HWN

PROFESSIONAL SEAL:

BGC ENGINEERING INC.
AN APPLIED EARTH SCIENCES COMPANY

CLIENT: SPANISH MOUNTAIN GOLD LTD.

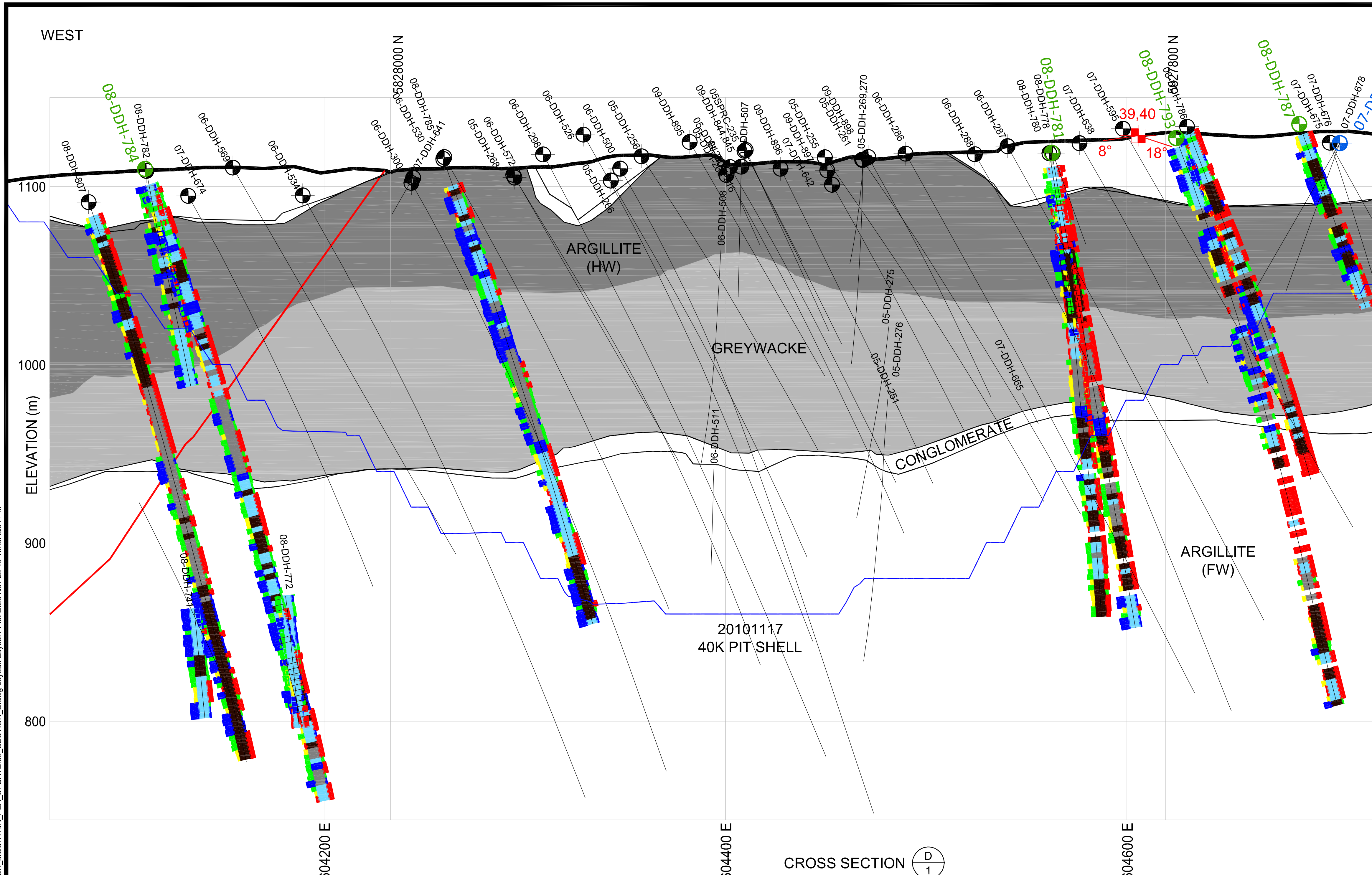
PROJECT: SPANISH MOUNTAIN PEA OPEN PIT SLOPE DESIGN

TITLE: GEOTECHNICAL SECTION C - MAIN ZONE

PROJECT No.:	DWG No.:	REV.:
0697-005	5	A

X:\Projects\0697\005_Spanish_Mtn\Workspace\20101012_REPORT_SPANISH_MOUNTAIN_PEA_UPDATE\05_SECTION_C.dwg Layout: Layout1 Plot Date Nov 25 10 Time: 3:03 PM

DWG TO BE READ WITH BGC REPORT TITLED "SPANISH MOUNTAIN PEA OPEN PIT SLOPE DESIGN" DATED NOVEMBER 26, 2010



CROSS SECTION D/1

Scale 1:2500

50 0 50 100 150

metres

NOTES:

1. PIT OUTLINE PROVIDED BY AGP MINING CONSULTANTS NOVEMBER 17, 2010.
2. CORE ORIENTATION INFORMATION COLLECTED BY SPANISH MOUNTAIN GOLD.
3. FRACTURES PER RUN AND RQD SHOWN ON DRILL HOLE BASED ON LOGGING CONDUCTED BY SPANISH MOUNTAIN GOLD.

LEGEND

- 51° BEDDING MEASUREMENT SHOWING APPARENT DIP [2]
- MAPPING STATION
- DRILL HOLE
- DRILL HOLE - LOGGED BY BGC
- DRILL HOLE - ORIENTED CORE
- TOPOGRAPHY
- 20101117 40K PIT SHELL
- FAULT
- ARGILLITE
- GREYWACKE
- OTHER LITHOLOGY CONTACT

DRILL HOLE ATTRIBUTES

LITHOLOGY - DOWN HOLE (ALONG CORE AXIS)

- FAULT ZONE
- MAFIC DYKE
- SILTSTONE
- ARGILLITE (HW)
- CONGLOMERATE
- GREYWACKE
- ARGILLITE (FW)

FRACTURES PER RUN - DOWN HOLE (RIGHT SIDE) [3]

- 0 - 20
- 20 - 40
- 40+

RQD PERCENT - DOWN HOLE (LEFT SIDE) [3]

- 0 - 25
- 25 - 50
- 50 - 75
- 75 - 125

REV.	DATE	REVISION NOTES	DRAWN	CHECK	APPR.

SCALE:	1:2500
DATE:	NOV 2010
DRAWN:	WKL
DESIGNED:	MK
CHECKED:	HWN
APPROVED:	HWN

PROFESSIONAL SEAL:

BGC ENGINEERING INC.
AN APPLIED EARTH SCIENCES COMPANY

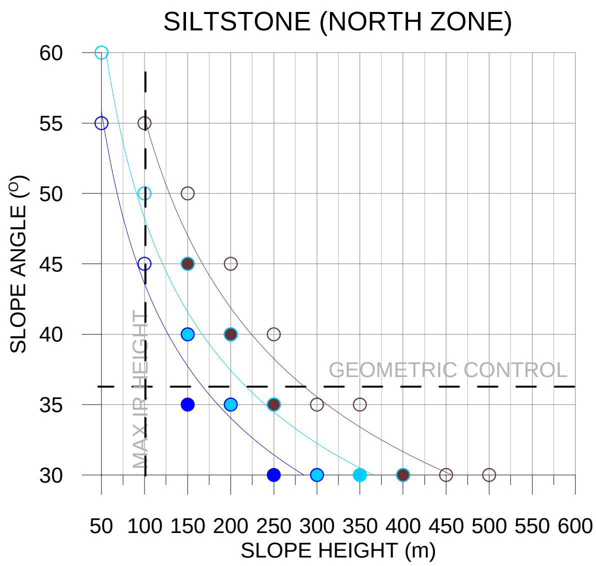
CLIENT: SPANISH MOUNTAIN GOLD LTD.

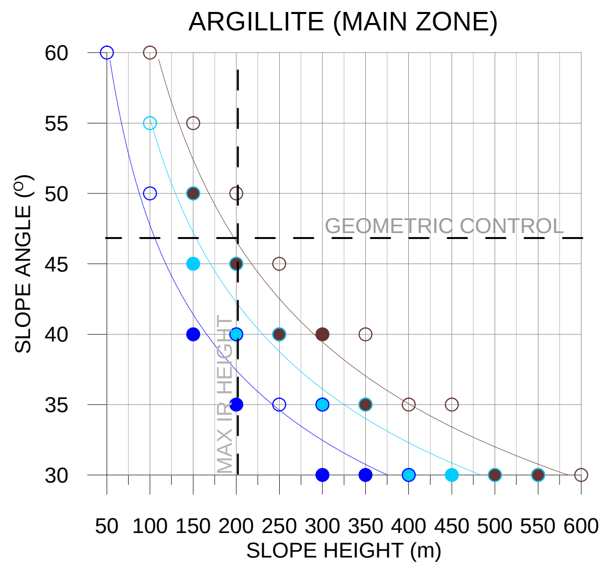
PROJECT: SPANISH MOUNTAIN PEA
OPEN PIT SLOPE DESIGN

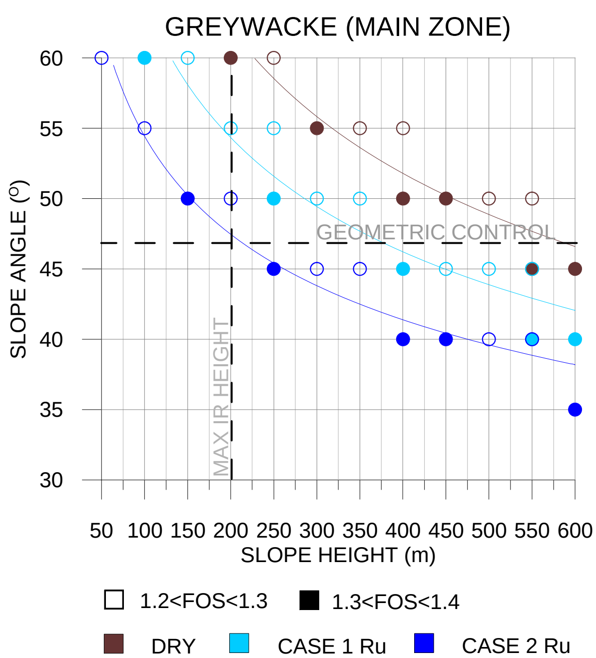
TITLE: GEOTECHINCAL SECTION D - MAIN ZONE

PROJECT No.:	DWG No.:	REV.:
0697-005	6	A

DWG TO BE READ WITH BGC REPORT TITLED "SPANISH MOUNTAIN PEA OPEN PIT SLOPE DESIGN" DATED NOVEMBER 26, 2010







HOEK-BROWN MATERIAL PARAMETERS USED FOR ANALYSES

Unit	GSI	Strength Grade	UCS (MPa)	Hoek-Brown Material Constant, mi	mb	s	Unit Weight (KN/m^3)
Gwk	57	3.0	25.0	18	1.246	0.0013	26
Arg	54	3.0	25.0	7	0.402	0.0008	26
Siltst	50	3.0	25.0	7	0.314	0.0004	26

NOTES:
1. The Hoek-Brown Criterion have been estimated using a disturbance factor ('D') of 0.85 for all units.
2. Mean RMR'76 parameters are used for each unit. GSI = RMR'76

NOTES:
1. RESULTS OF GENERIC NON-CIRCULAR SLOPE ANALYSES WHERE THE FACTOR OF SAFETY FALLS INTO THE RANGE 1.3±0.1 ARE PLOTTED.
2. CASE 1 Ru = PORE WATER COEFFICIENT EQUIVALENT TO A WATER COLUMN THAT IS 25% THE HEIGHT OF THE ROCK COLUMN.
3. CASE 2 Ru = PORE WATER COEFFICIENT EQUIVALENT TO A WATER COLUMN THAT IS 50% THE HEIGHT OF THE ROCK COLUMN.
4. CURVES SHOWN REPRESENT POTENTIAL ROCK MASS CONTROLLED FAILURES, NOT FINAL DESIGN ANGLES.
5. SEE TABLE 3 FOR FINAL DESIGN CRITERIA.

SCALE:	AS SHOWN	DESIGNED:	JW
DATE:	NOV 2010	CHECKED:	HWN
DRAWN:	JW	APPROVED:	HWN

BGC ENGINEERING INC.
AN APPLIED EARTH SCIENCES COMPANY

PROJECT:
SPANISH MOUNTAIN PEA
OPEN PIT SLOPE DESIGN

TITLE:
GENERIC STABILITY ANALYSIS RESULTS

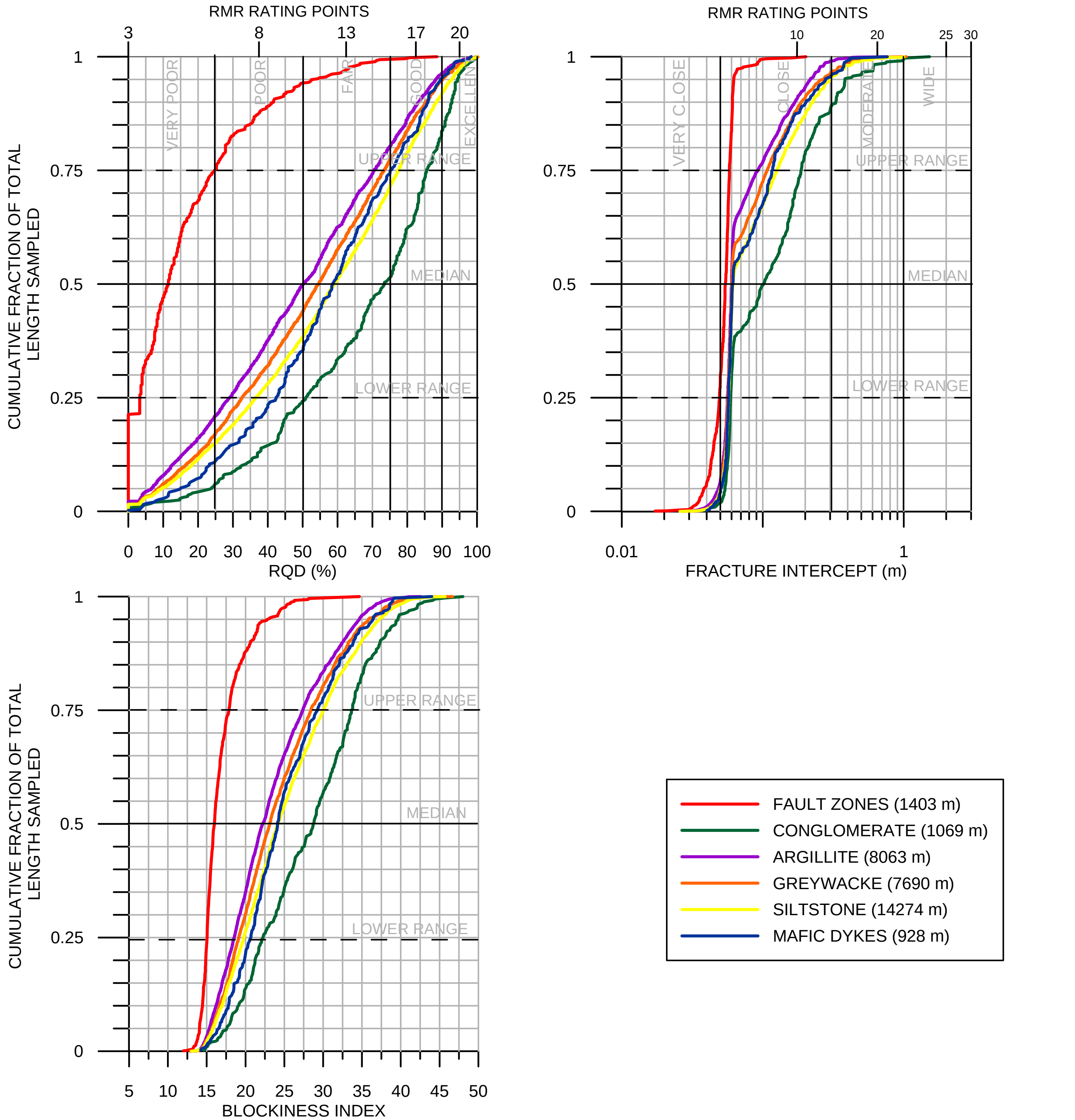
CLIENT:
SPANISH MOUNTAIN GOLD LTD.

PROJECT No.:	DWG No.:	REV.:
0697-005	7	0

DWG. TO BE READ WITH BGC REPORT TITLED "SPANISH MOUNTAIN PEA, ENGINEERING GEOLOGY, AND GEOTECHNICAL EVALUATIONS", DATED NOV 2010.

APPENDIX A

Field Data and Geomechanical Summary Logs



NOTES
1. DATA WAS COLLECTED BY SPANISH MOUNTAIN GOLD LTD.
2. QUALITATIVE DESCRIPTIONS HAVE BEEN SHOWN FOR RQD (DEERE AND DEERE, 1989) AND FRACTURE INTERCEPT (ISRM, 1981).
3. BLOCKINESS INDEX IS THE SUM OF RMR '76 RATING POINTS FOR RQD AND FRACTURE INTERCEPT.
4. SEE TABLE 1 FOR DESIGN ROCK MASS PROPERTIES ASSIGNED TO EACH UNIT.

SCALE:	AS SHOWN	DESIGNED:	JW
DATE:	NOV 2010	CHECKED:	HWN
DRAWN:	JW	APPROVED:	HWN

BGC ENGINEERING INC.
AN APPLIED EARTH SCIENCES COMPANY

PROJECT:
SPANISH MOUNTAIN PEA
OPEN PIT SLOPE DESIGN

TITLE:
DISTRIBUTION OF RQD,
FRACTURE INTERCEPT, AND BLOCKINESS INDEX

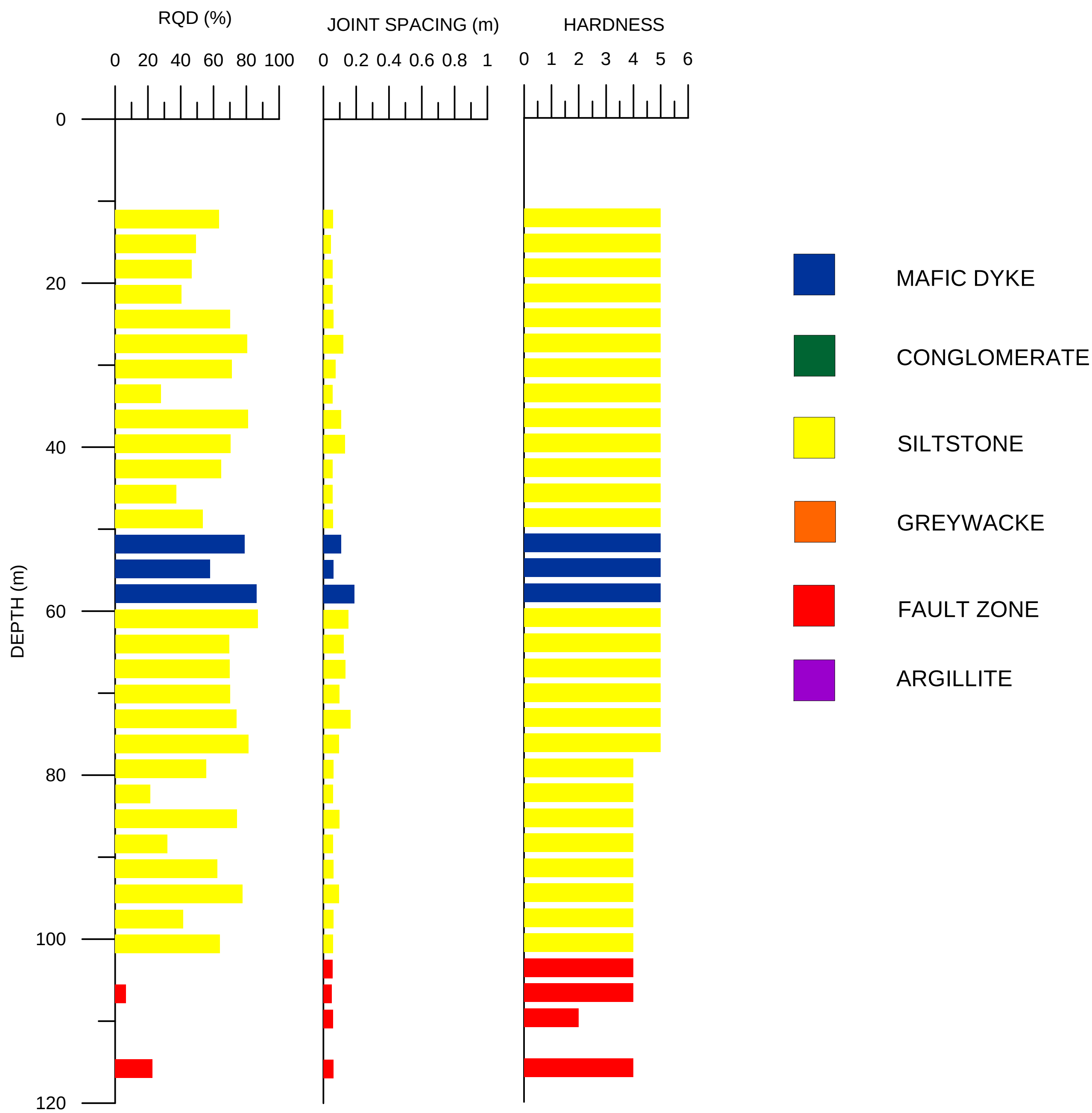
CLIENT:
SPANISH MOUNTAIN GOLD LTD.

PROJECT No.:	DWG No.:	REV.:
0697-005	A1	0

K:\Data\Projects\0697 PEG Mining\005 Spanish Mountain Gold\2005-09 drill logs\Cumulative Frequency Plots

DWG. TO BE READ WITH BGC REPORT TITLED "SPANISH MOUNTAIN PEA, ENGINEERING GEOLOGY, AND GEOTECHNICAL EVALUATIONS", DATED NOV. 2010.

08-DDH-731



NOTES:
1. DATA WAS COLLECTED BY SPANISH MOUNTAIN GOLD LTD. FROM 2008 DRILLHOLES.

SCALE:	AS SHOWN	DESIGNED:	JW
DATE:	NOV 2010	CHECKED:	HWN
DRAWN:	JW	APPROVED:	HWN

BGC ENGINEERING INC.
AN APPLIED EARTH SCIENCES COMPANY

PROJECT: SPANISH MOUNTAIN PEA
OPEN PIT SLOPE DESIGN

TITLE: RMR '76 PARAMETERS

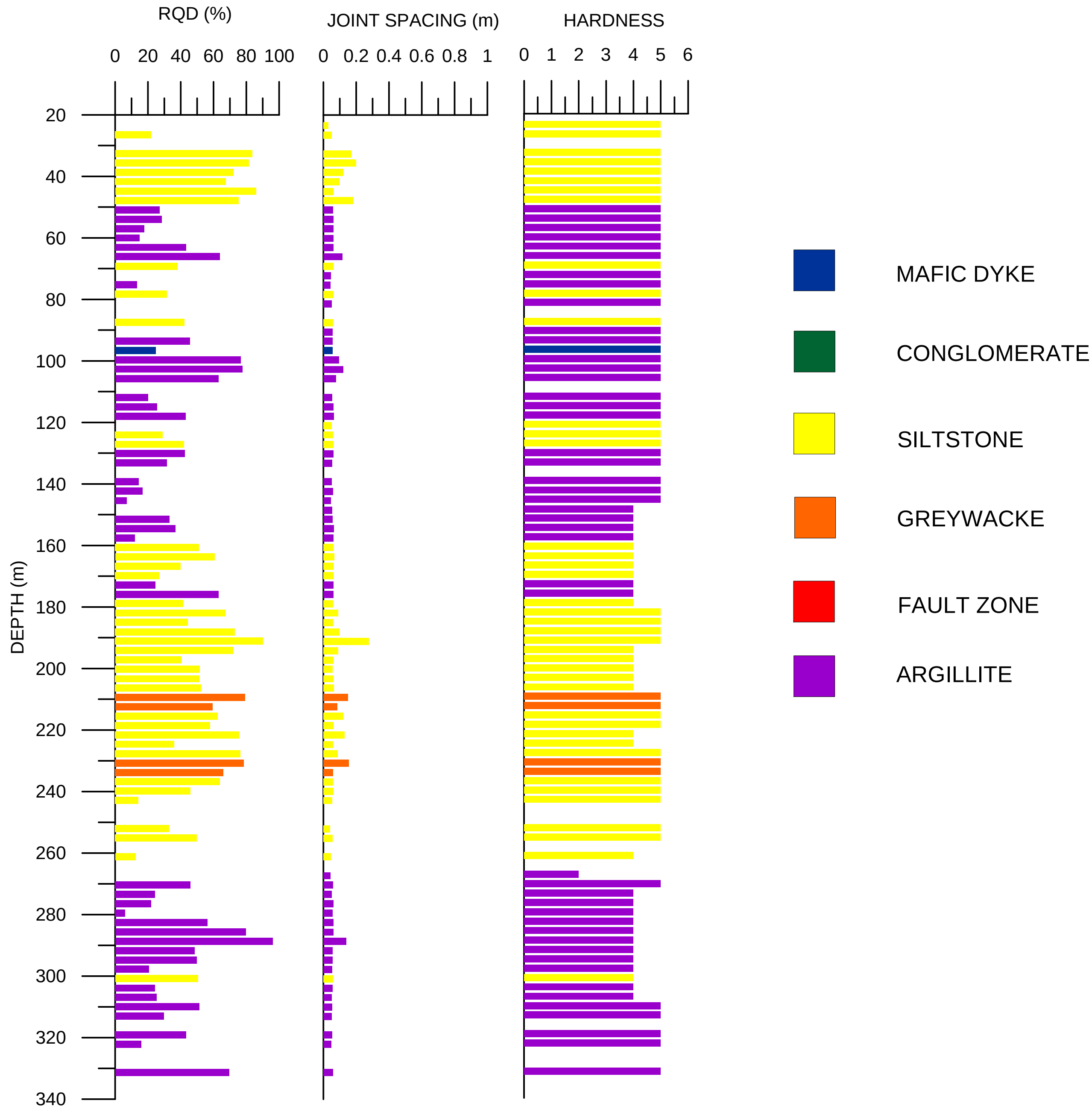
CLIENT: SPANISH MOUNTAIN GOLD LTD.

PROJECT No.:	DWG No.:	REV.:
0697-005	A2	0

K:\Data\Projects\0697 PEG Mining\005 Spanish Mountain Gold\2005-09 drill logs\RMR Logs

DWG. TO BE READ WITH BGC REPORT TITLED "SPANISH MOUNTAIN PEA, ENGINEERING GEOLOGY, AND GEOTECHNICAL EVALUATIONS", DATED NOV. 2010.

08-DDH-746



NOTES:
1. DATA WAS COLLECTED BY SPANISH MOUNTAIN GOLD LTD. FROM 2008 DRILLHOLES.

SCALE:	AS SHOWN	DESIGNED:	JW
DATE:	NOV 2010	CHECKED:	HWN
DRAWN:	JW	APPROVED:	HWN

BGC ENGINEERING INC.
AN APPLIED EARTH SCIENCES COMPANY

PROJECT: SPANISH MOUNTAIN PEA
OPEN PIT SLOPE DESIGN

TITLE: RMR '76 PARAMETERS

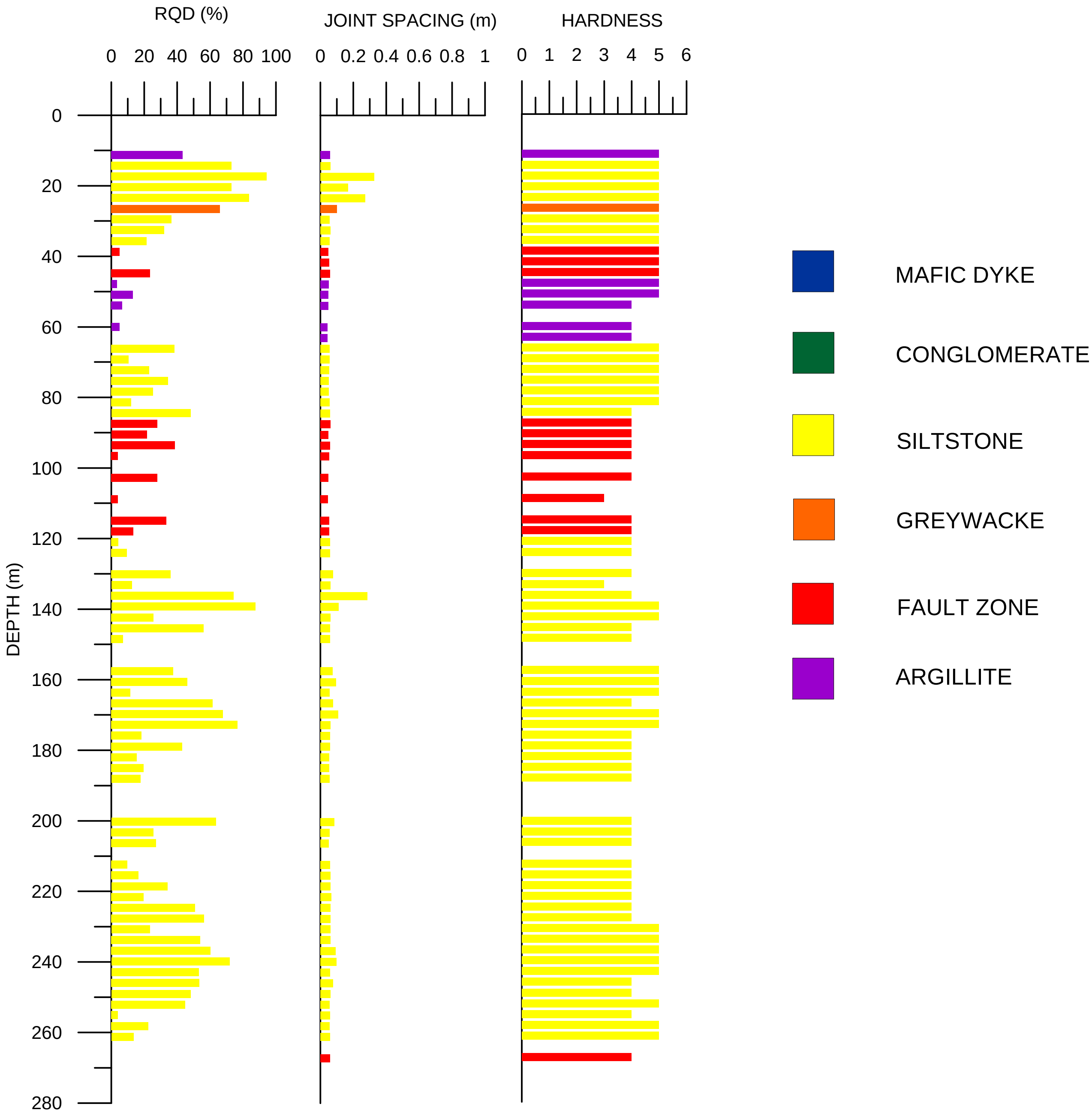
CLIENT: SPANISH MOUNTAIN GOLD LTD.

PROJECT No.:	DWG No.:	REV.:
0697-005	A3	0

K:\Data\Projects\0697 PEG Mining\005 Spanish Mountain Gold\2005-09 drill logs\RMR Logs

DWG. TO BE READ WITH BGC REPORT TITLED "SPANISH MOUNTAIN PEA, ENGINEERING GEOLOGY, AND GEOTECHNICAL EVALUATIONS", DATED NOV. 2010.

08-DDH-766



NOTES:
1. DATA WAS COLLECTED BY SPANISH MOUNTAIN GOLD LTD. FROM 2008 DRILLHOLES.

SCALE:	AS SHOWN	DESIGNED:	JW
DATE:	NOV 2010	CHECKED:	HWN
DRAWN:	JW	APPROVED:	HWN

BGC ENGINEERING INC.
AN APPLIED EARTH SCIENCES COMPANY

PROJECT:
SPANISH MOUNTAIN PEA
OPEN PIT SLOPE DESIGN

TITLE:
RMR '76 PARAMETERS

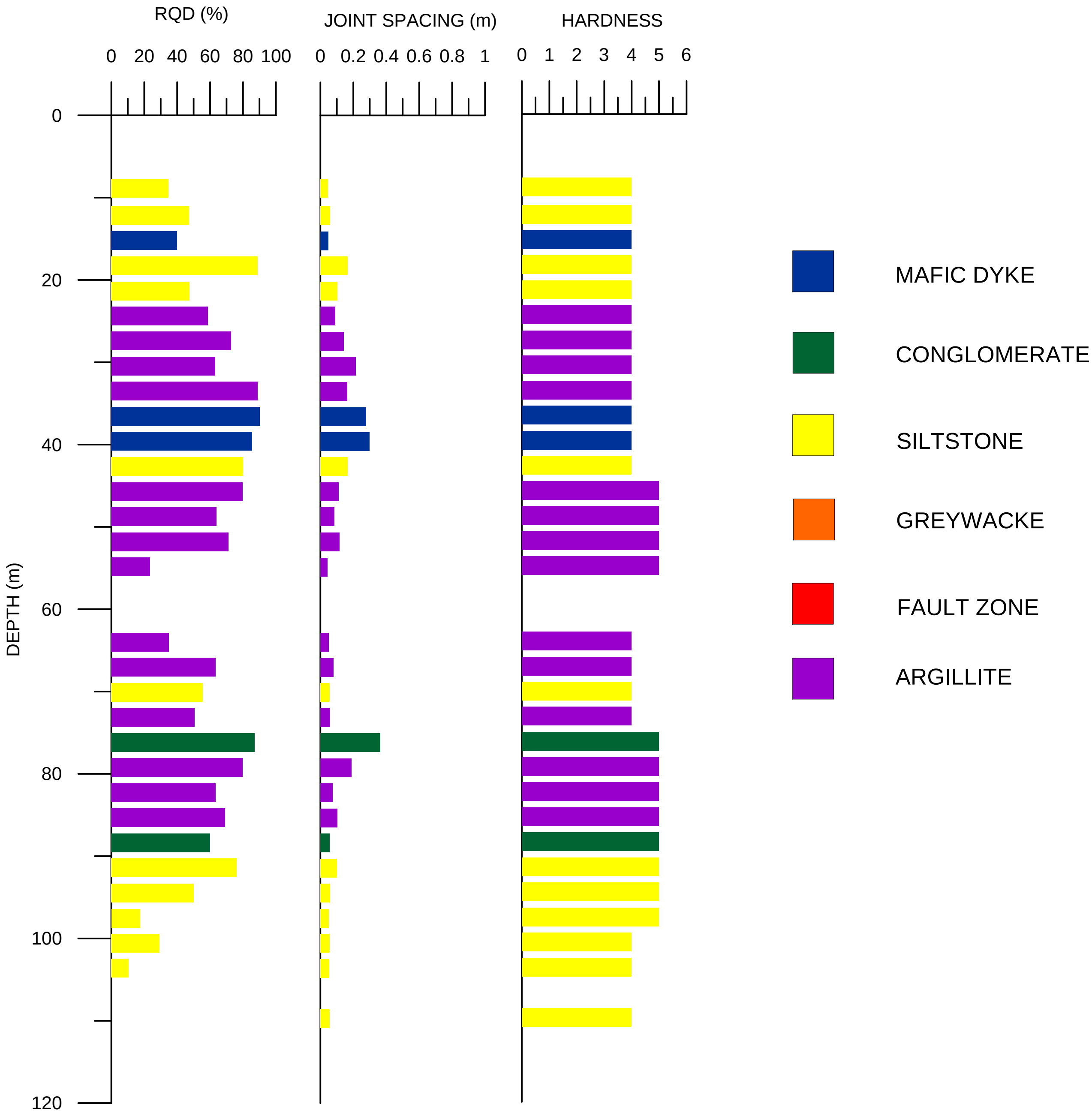
CLIENT:
SPANISH MOUNTAIN GOLD LTD.

PROJECT No.:	DWG No.:	REV.:
0697-005	A4	0

DWG. TO BE READ WITH BGC REPORT TITLED "SPANISH MOUNTAIN PEA, ENGINEERING GEOLOGY, AND GEOTECHNICAL EVALUATIONS", DATED NOV. 2010.

K:\Data\Projects\0697 PEG Mining\005 Spanish Mountain Gold\2005-09 drill logs\RMR Logs

08-DDH-787



NOTES:
1. DATA WAS COLLECTED BY SPANISH MOUNTAIN GOLD LTD. FROM 2008 DRILLHOLES.

SCALE:	AS SHOWN	DESIGNED:	JW
DATE:	NOV 2010	CHECKED:	HWN
DRAWN:	JW	APPROVED:	HWN

BGC ENGINEERING INC.
AN APPLIED EARTH SCIENCES COMPANY

PROJECT: SPANISH MOUNTAIN PEA OPEN PIT SLOPE DESIGN

TITLE: RMR '76 PARAMETERS

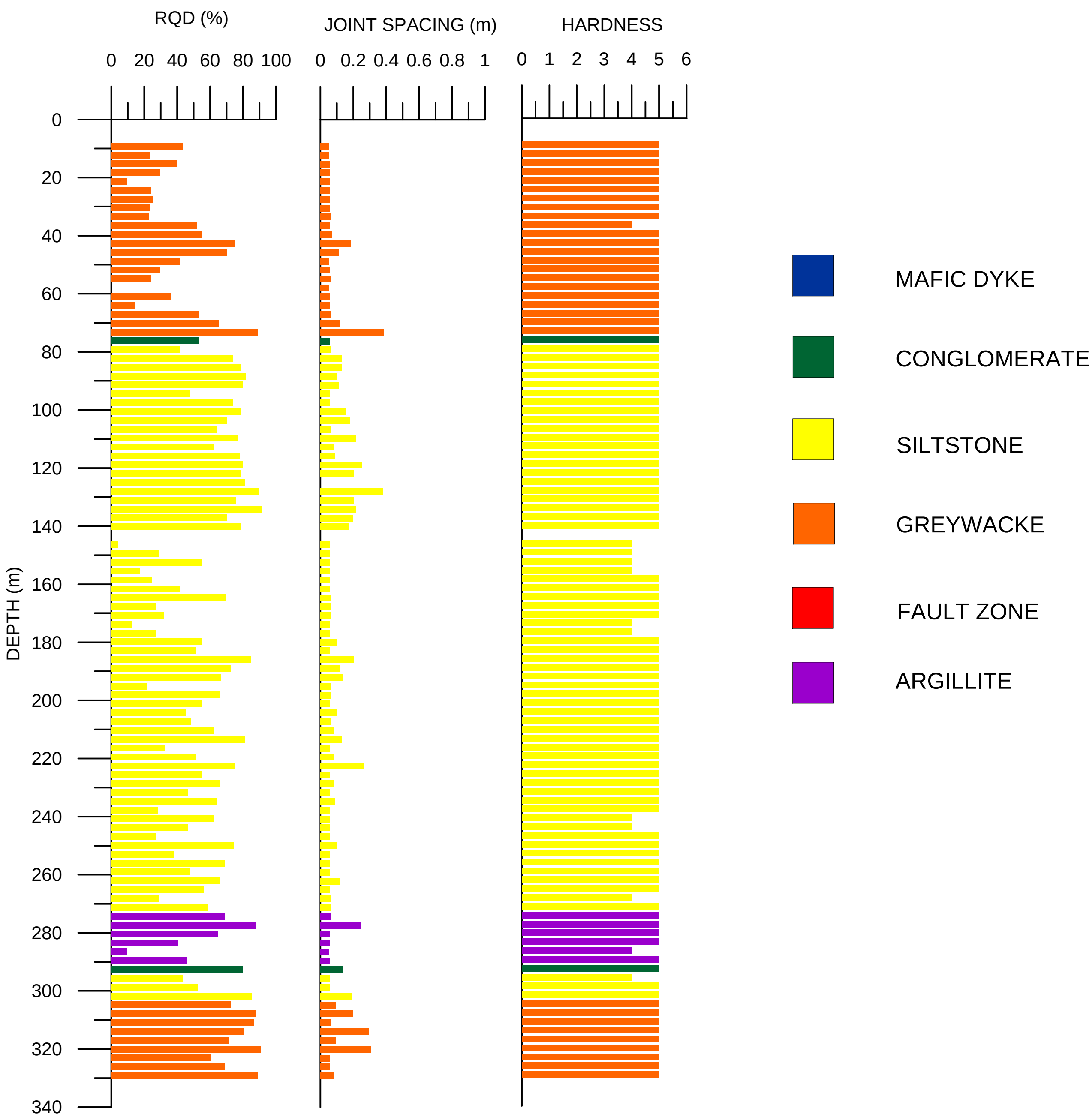
CLIENT: SPANISH MOUNTAIN GOLD LTD.

PROJECT No.:	DWG No.:	REV.:
0697-005	A5	0

K:\Data\Projects\0697 PEG Mining\005 Spanish Mountain Gold\2005-09 drill logs\RMR Logs

DWG. TO BE READ WITH BGC REPORT TITLED "SPANISH MOUNTAIN PEA, ENGINEERING GEOLOGY, AND GEOTECHNICAL EVALUATIONS", DATED NOV. 2010.

08-DDH-796



NOTES:
1. DATA WAS COLLECTED BY SPANISH MOUNTAIN GOLD LTD. FROM 2008 DRILLHOLES.

SCALE:	AS SHOWN	DESIGNED:	JW
DATE:	NOV 2010	CHECKED:	HWN
DRAWN:	JW	APPROVED:	HWN

BGC ENGINEERING INC.
AN APPLIED EARTH SCIENCES COMPANY

PROJECT: SPANISH MOUNTAIN PEA
OPEN PIT SLOPE DESIGN

TITLE: RMR '76 PARAMETERS

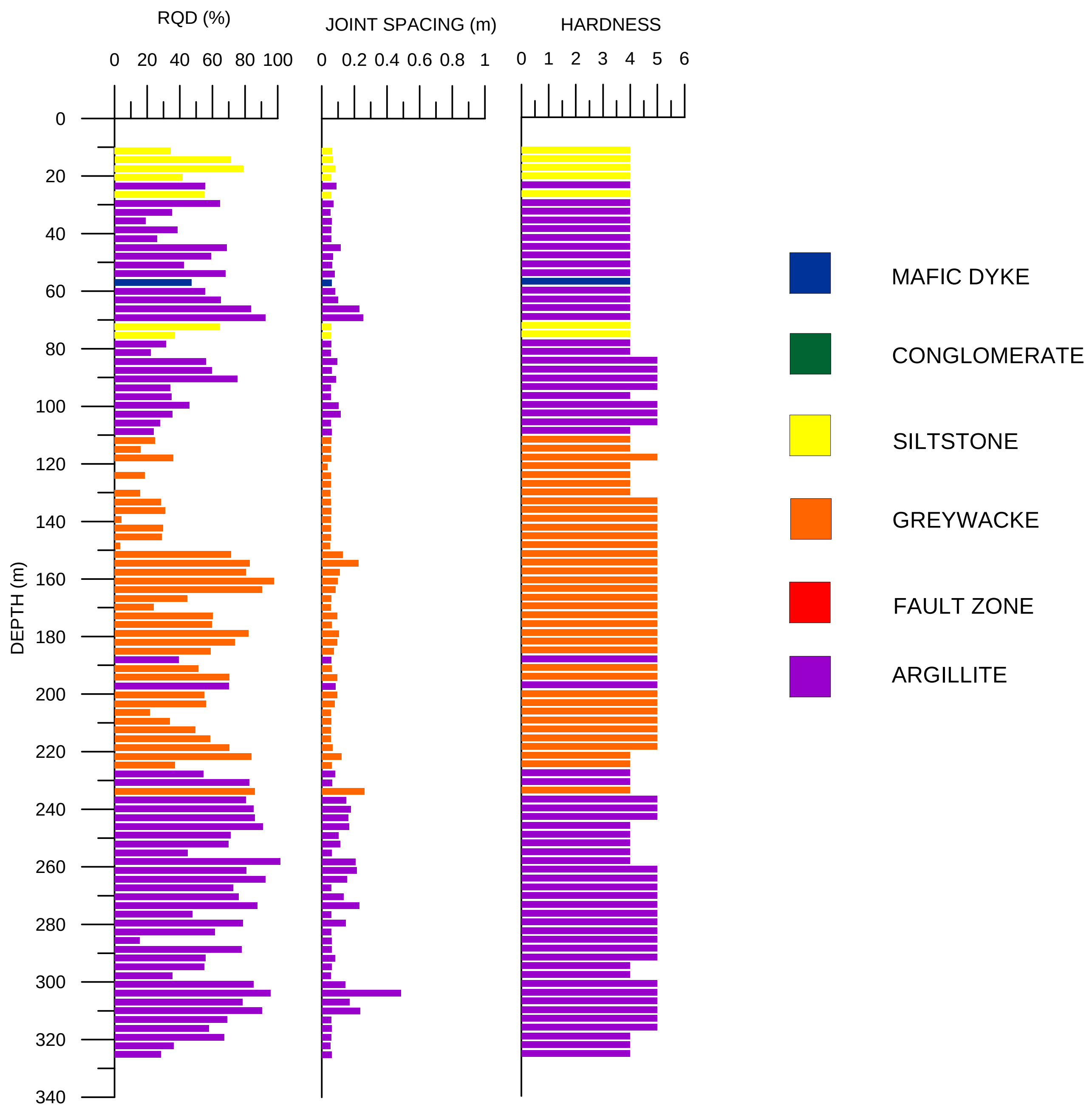
CLIENT: SPANISH MOUNTAIN GOLD LTD.

PROJECT No.:	DWG No.:	REV.:
0697-005	A6	0

K:\Data\Projects\0697 PEG Mining\005 Spanish Mountain Gold\2005-09 drill logs\RMR Logs

DWG. TO BE READ WITH BGC REPORT TITLED "SPANISH MOUNTAIN PEA, ENGINEERING GEOLOGY, AND GEOTECHNICAL EVALUATIONS", DATED NOV. 2010.

08-DDH-807



NOTES:
1. DATA WAS COLLECTED BY SPANISH MOUNTAIN GOLD LTD. FROM 2008 DRILLHOLES.

SCALE:	AS SHOWN	DESIGNED:	JW
DATE:	NOV 2010	CHECKED:	HWN
DRAWN:	JW	APPROVED:	HWN

BGC ENGINEERING INC.
AN APPLIED EARTH SCIENCES COMPANY

PROJECT:
SPANISH MOUNTAIN PEA
OPEN PIT SLOPE DESIGN

TITLE:
RMR '76 PARAMETERS

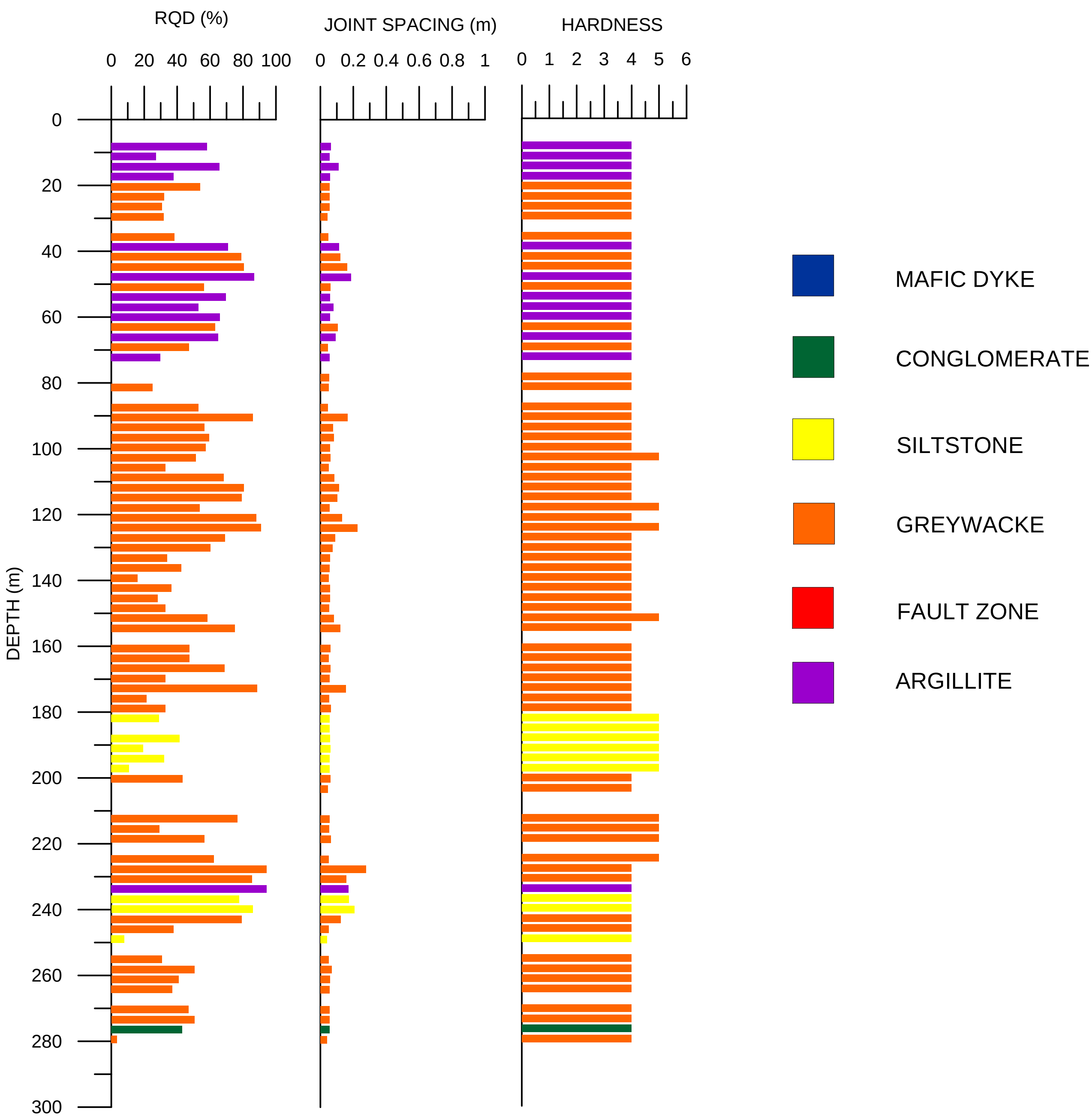
CLIENT:
SPANISH MOUNTAIN GOLD LTD.

PROJECT No.:	DWG No.:	REV.:
0697-005	A7	0

K:\Data\Projects\0697 PEG Mining\005 Spanish Mountain Gold\2005-09 drill logs\RMR Logs

DWG. TO BE READ WITH BGC REPORT TITLED "SPANISH MOUNTAIN PEA, ENGINEERING GEOLOGY, AND GEOTECHNICAL EVALUATIONS", DATED NOV. 2010.

08-DDH-813



NOTES:
1. DATA WAS COLLECTED BY SPANISH MOUNTAIN GOLD LTD. FROM 2008 DRILLHOLES.

SCALE:	AS SHOWN	DESIGNED:	JW
DATE:	NOV 2010	CHECKED:	HWN
DRAWN:	JW	APPROVED:	HWN

BGC ENGINEERING INC.
AN APPLIED EARTH SCIENCES COMPANY

PROJECT:
SPANISH MOUNTAIN PEA
OPEN PIT SLOPE DESIGN

TITLE:
RMR '76 PARAMETERS

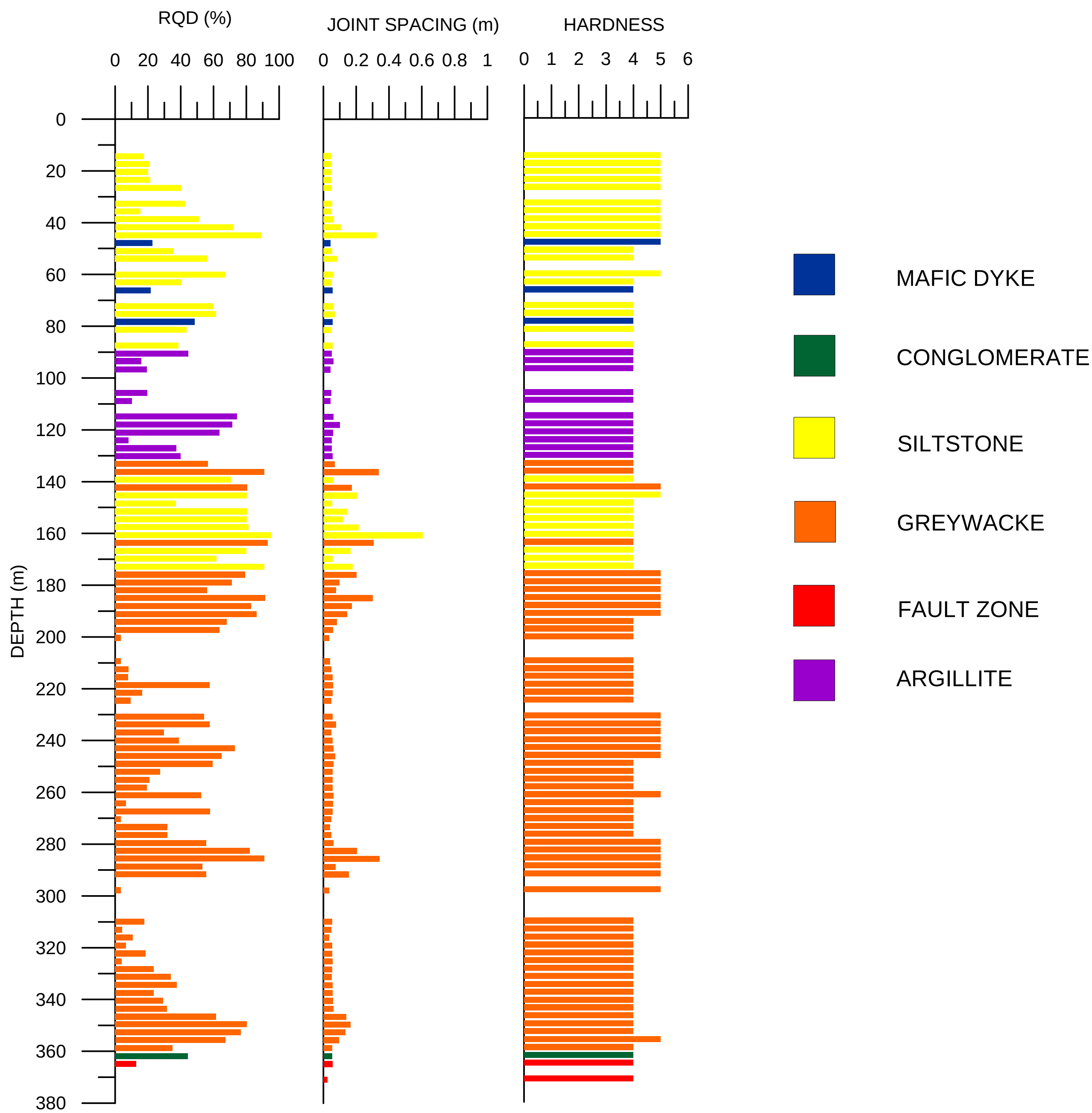
CLIENT:
SPANISH MOUNTAIN GOLD LTD.

PROJECT No.:	DWG No.:	REV.:
0697-005	A8	0

K:\Data\Projects\0697 PEG Mining\005 Spanish Mountain Gold\2005-09 drill logs\RMR Logs

DWG. TO BE READ WITH BGC REPORT TITLED "SPANISH MOUNTAIN PEA, ENGINEERING GEOLOGY, AND GEOTECHNICAL EVALUATIONS", DATED NOV. 2010.

08-DDH-819



NOTES:
1. DATA WAS COLLECTED BY SPANISH MOUNTAIN GOLD LTD. FROM 2008 DRILLHOLES.

SCALE:	AS SHOWN	DESIGNED:	JW
DATE:	NOV 2010	CHECKED:	HWN
DRAWN:	JW	APPROVED:	HWN

BGC ENGINEERING INC.
AN APPLIED EARTH SCIENCES COMPANY

PROJECT: SPANISH MOUNTAIN PEA
OPEN PIT SLOPE DESIGN

TITLE: RMR '76 PARAMETERS

CLIENT: SPANISH MOUNTAIN GOLD LTD.

PROJECT No.:	DWG No.:	REV.:
0697-005	A9	0

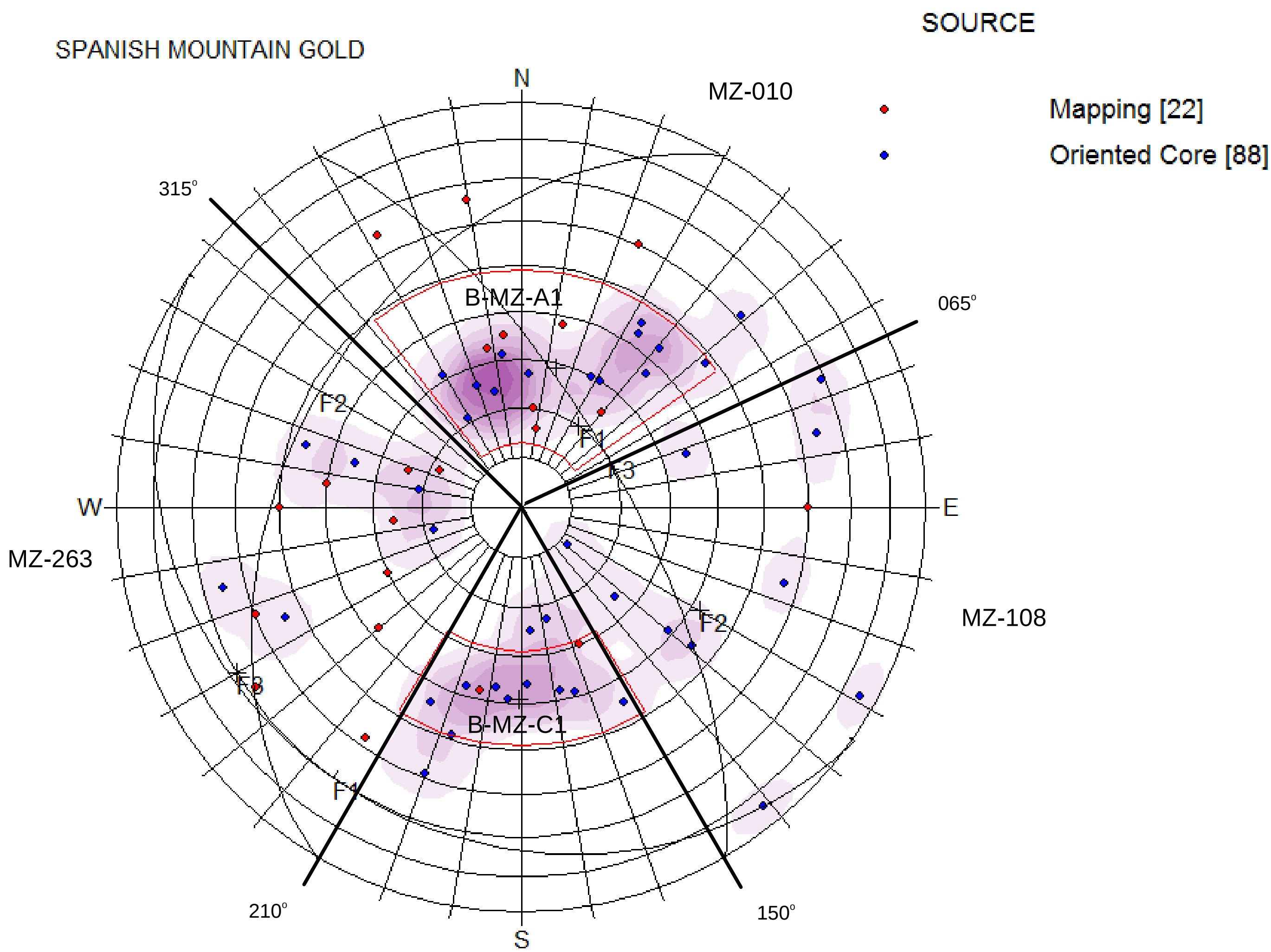
K:\Data\Projects\0697 PEG Mining\005 Spanish Mountain Gold\2005-09 drill logs\RMR Logs

DWG. TO BE READ WITH BGC REPORT TITLED "SPANISH MOUNTAIN PEA, ENGINEERING GEOLOGY, AND GEOTECHNICAL EVALUATIONS", DATED NOV. 2010.

APPENDIX B

Structural Geologic Information and Rock Mass Fabric Stereonets

SPANISH MOUNTAIN GOLD



MAIN ZONE BEDDING

SET	DIP	DIP DIRECTION
B-MZ-A1	29°	194°
B-MZ-C1	39°	001°

NOTES:
1.STRUCTURAL DISCONTINUITY MEASUREMENTS GATHERED FROM SURFICIAL MAPPING CONDUCTED BY/FOR SKYLINE GOLD.. ORIENTED DRILLHOLE DATA PROVIDED BY SPANISH MOUNTAIN GOLD LTD. MAY 2010

SCALE:	AS SHOWN	DESIGNED:	HWN
DATE:	NOV 2010	CHECKED:	HWN
DRAWN:	MK	APPROVED:	HWN

BGC ENGINEERING INC.
AN APPLIED EARTH SCIENCES COMPANY

PROJECT:
SPANISH MOUNTAIN GOLD PEA
OPEN PIT SLOPE DESIGN

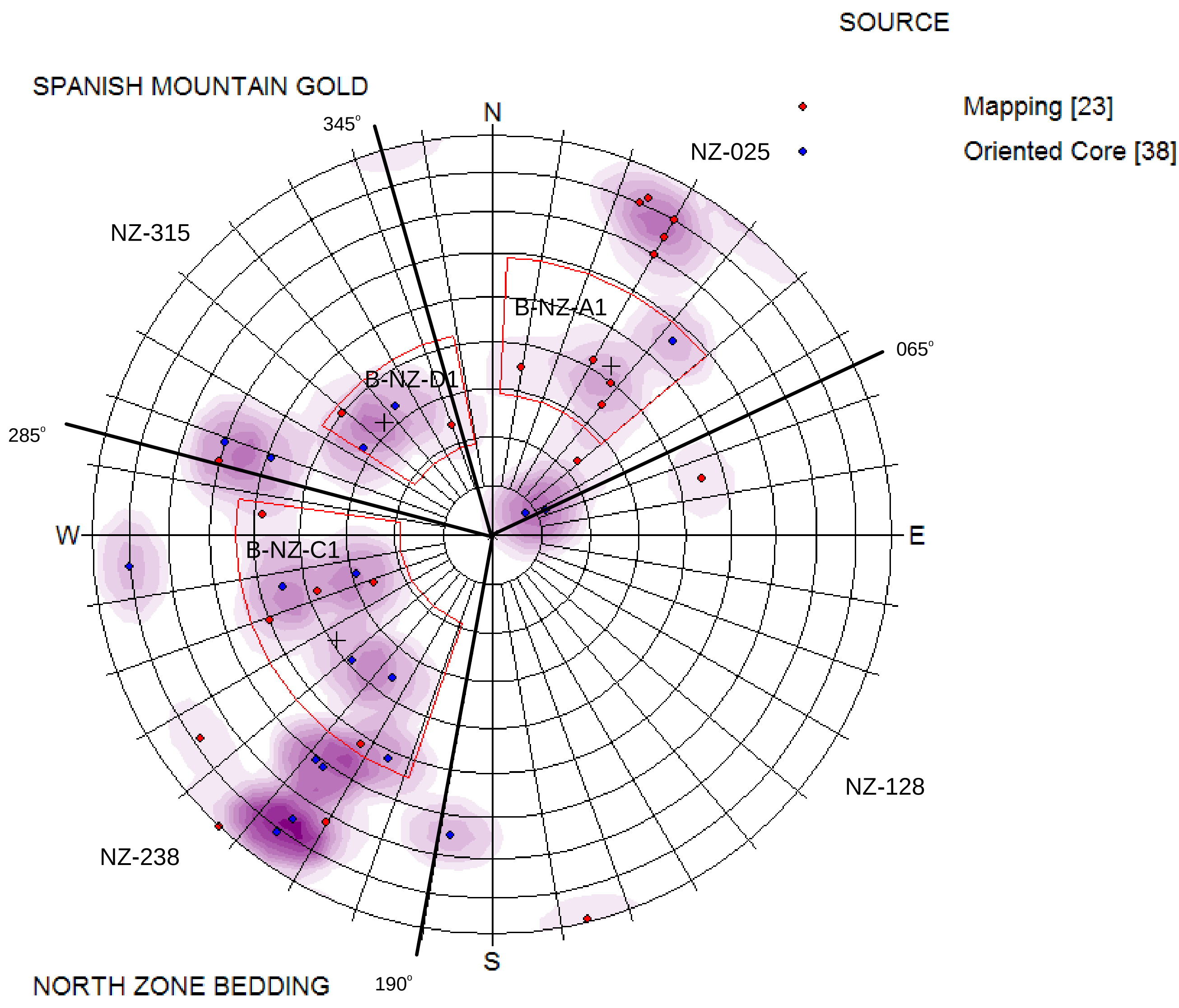
TITLE:
STRUCTURAL DISCONTINUITY ORIENTATION MEASUREMENTS
MAIN ZONE

CLIENT:
SPANISH MOUNTAIN GOLD LTD.

PROJECT No.:	DWG No.:	REV.:
0697-005	B1	A

K:\Data\Projects\0697 PEG Mining\005 Spanish Mountain Gold\02 Open Pit PEA\04 Analysis\Structural\Oriented core and mapping\Stereonets

DWG. TO BE READ WITH BGC REPORT TITLED "SPANISH MOUNTAIN PEA, ENGINEERING GEOLOGY, AND GEOTECHNICAL EVALUATIONS", DATED NOV. 2010.



SET	DIP	DIP DIRECTION
B-NZ-A1	43°	215°
B-NZ-C1	39°	056°
B-NZ-D1	32°	136°

NOTES:
1.STRUCTURAL DISCONTINUITY MEASUREMENTS GATHERED FROM SURFICIAL MAPPING CONDUCTED BY/FOR SYKLINE GOLD. ORIENTED DRILLHOLE DATA PROVIDED BY SPANISH MOUNTAIN GOLD LTD. MAY 2010

SCALE:	AS SHOWN	DESIGNED:	HWN
DATE:	NOV 2010	CHECKED:	HWN
DRAWN:	MK	APPROVED:	HWN

BGC ENGINEERING INC.
AN APPLIED EARTH SCIENCES COMPANY

PROJECT:
SPANISH MOUNTAIN GOLD PEA
OPEN PIT SLOPE DESIGN

TITLE:
STRUCTURAL DISCONTINUITY ORIENTATION MEASUREMENTS
NORTH ZONE

CLIENT:
SPANISH MOUNTAIN GOLD LTD.

PROJECT No.:	DWG No.:	REV.:
0697-005	B2	A

K:\Data\Projects\0697 PEG Mining\005 Spanish Mountain Gold\02 Open Pit PEA\04 Analysis\Structural\Oriented core and mapping\Stereonets

DWG. TO BE READ WITH BGC REPORT TITLED "SPANISH MOUNTAIN PEA, ENGINEERING GEOLOGY, AND GEOTECHNICAL EVALUATIONS", DATED NOV. 2010.

APPENDIX D

Production Rate Trade-Off Study
Mine Schedule with the Associated Waste Dump Allocation

Spanish Mountain Gold

Spanish Mountain

Notes:

All cells with green shading are adjustable and will be reflected in the appropriate tables and calculations
Unless otherwise noted, all dollars are denoted in Canadian currency

COMMODITY PRICE SCENARIOS

		Engineering Base	Financial Base
Gold Price	$US/ounce	$950.00	$1,298.00
Gold Refining Charge	$US/ounce	$8.00	$8.00
Gold Refinery Payable	%	98.5%	98.5%
Net Gold Price	$US/ounce	$927.75	$1,270.53

Exchange Rate	$Cdn:$US	1.10	1.03

Gold Price	$Cdn/ounce	$1,045.00	$1,336.94
Gold Refining Charge	$Cdn/ounce	$8.80	$8.24
Gold Refinery Payable	%	98.5%	98.5%
Net Gold Price	$Cdn/ounce	$1,020.53	$1,308.65

Production Rate Options

		Plant Production Rate - Ore tonnes per day						
		20,000	**25,000**	**30,000**	**35,000**	**40,000**	**50,000**	**40,000 M**
Open Pit Mining Cost - Base Rate	$/tonne	$2.15	$2.08	$2.00	$1.94	$1.89	$1.84	$1.89
Adjustment factor	%	100%	100%	100%	100%	100%	100%	100%
Adjusted mining cost	$/tonne	$2.15	$2.08	$2.00	$1.94	$1.89	$1.84	$1.89
Processing Cost	$/tonne ore	$5.92	$5.67	$5.50	$5.34	$5.27	$5.20	$5.27
Tailings Cost	$/tonne ore	$0.16	$0.16	$0.16	$0.16	$0.16	$0.16	$0.16
General & Administrative	$/tonne ore	$0.70	$0.59	$0.56	$0.54	$0.52	$0.42	$0.52

Owners Cost	$	$0

Average Insitu Grade	gram/tonne	0.60	0.57	0.56	0.53	0.53	0.53	0.58
Dilution	%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Average Diluted Grade	gram/tonne	0.60	0.57	0.56	0.53	0.53	0.53	0.58

Recovery

Gold		**Gold Grade >**	**Recovery**	
Gold Grade	grams/tonne	0.50	90.0%	
Gold Grade	grams/tonne	0.30	90.0%	88
Gold Grade	grams/tonne	0.00	90.0%	86

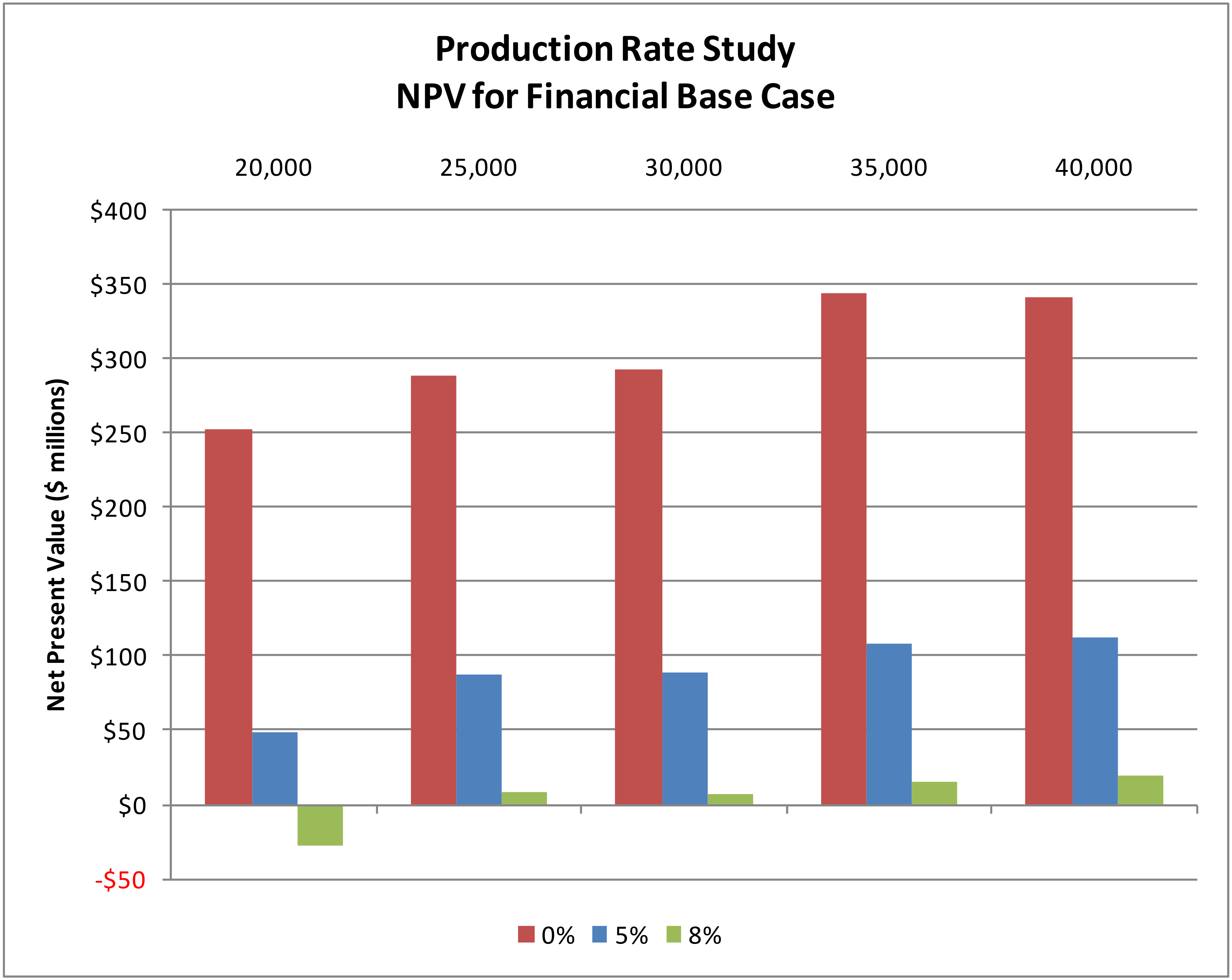
Royalty	%	1.0%

Adjustment Factors

	Capital Dollars	Indirects	Contingency
OP Mining	100%	10.0%	15.0%
UG Mining	100%	0.0%	15.0%
Processing	100%	30.4%	15.0%
Infrastructure	100%	22.0%	15.0%
Environmental	100%	15.0%	15.0%

Economic Evaluation Results

	Mining Process Rate	**Net Present Value ($ millions)**			**Internal**	**Net Revenue**	**Total Operating**	**Capital Cost ($ Millions)**							
Metal Price Option	**tonnes/day**	**0%**	**5%**	**8%**	**Rate of Return**	**less Royalty**	**Cost**	**Total**	**OP Mining**	**UG Mining**	**Processing**	**Infrastructure**	**Environmental**	**Indirects**	**Contingency**
Engineering Base	20,000	-$146	-$220	-$245	-5%	**$1,411**	**$994**	**$562.7**	$83.9	$0.0	$167.7	$140.9	$16.2	$92.8	$61.3
	25,000	-$133	-$207	-$233	-5%	**$1,494**	**$1,031**	**$595.8**	$79.9	$0.0	$193.8	$140.9	$16.2	$100.3	$64.6
	30,000	-$155	-$229	-$256	-6%	**$1,583**	**$1,088**	**$650.4**	$95.2	$0.0	$218.3	$140.9	$16.2	$109.3	$70.6
	35,000	-$193	-$268	-$294	-6%	**$1,902**	**$1,382**	**$713.0**	$118.5	$0.0	$241.2	$140.9	$16.2	$118.6	$77.5
	40,000	-$211	-$283	-$308	-7%	**$1,957**	**$1,412**	**$755.9**	$127.4	$0.0	$263.1	$140.9	$16.2	$126.1	$82.1
Financial Base	20,000	$252	$49	-$28	7%	**$1,809**	**$994**	**$562.7**	$83.9	$0.0	$167.7	$140.9	$16.2	$92.8	$61.3
	25,000	$289	$87	$8	8%	**$1,916**	**$1,031**	**$595.8**	$79.9	$0.0	$193.8	$140.9	$16.2	$100.3	$64.6
	30,000	$292	$88	$6	8%	**$2,030**	**$1,088**	**$650.4**	$95.2	$0.0	$218.3	$140.9	$16.2	$109.3	$70.6
	35,000	$344	$108	$15	9%	**$2,439**	**$1,382**	**$713.0**	$118.5	$0.0	$241.2	$140.9	$16.2	$118.6	$77.5
	40,000	$342	$112	$20	9%	**$2,510**	**$1,412**	**$755.9**	$127.4	$0.0	$263.1	$140.9	$16.2	$126.1	$82.1



Production Rate Study
NPV for Financial Base Case
20,000
25,000
30,000
35,000
40,000
$400
$350
$300
$250
$200
$150
$100
$50
$0
-$50
Net Present Value ($ millions)
0%
5%
8%

20K Cashflow

Engineering Base

Plant Throughput	**20,000**	**tonnes per day**																
		Total		**Year -2**	**Year -1**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	**Year 11**	**Year 12**	**Year 13**
Mill Production																		
Mill Feed	tonnes	80,705,097		-	-	5,400,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	3,305,097	-
Gold Grade	g/t	0.60		-	-	0.48	0.50	0.66	0.79	0.54	0.66	0.72	0.60	0.53	0.59	0.53	0.48	-
Gold Recovery	%			0.0%	0.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	0.0%
Insitu Gold	ounces	1,551,303		-	-	84,058	115,565	153,835	182,700	123,854	153,205	165,625	138,145	122,675	137,665	122,470	51,507	-
Recovered Gold	ounces	1,396,173		-	-	75,652	104,009	138,451	164,430	111,469	137,885	149,063	124,330	110,407	123,899	110,223	46,356	-
Mine Production																		
Open Pit																		
Ore																		
Ore to Mill	tonnes	80,028,004		-	-	4,722,907	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	3,305,097	-
Ore to Stockpile	tonnes	677,093		-	677,093	-	-	-	-	-	-	-	-	-	-	-	-	-
Stockpile to Mill	tonnes	677,093		-	-	677,093	-	-	-	-	-	-	-	-	-	-	-	-
Waste	tonnes	124,563,443		-	4,038,543	9,206,311	10,908,793	11,292,373	12,111,315	12,301,289	11,300,840	12,642,345	19,168,064	9,749,014	5,811,564	5,429,385	603,609	-
Total Material	tonnes	205,945,633		-	4,715,636	14,606,311	18,108,793	18,492,373	19,311,315	19,501,289	18,500,840	19,842,345	26,368,064	16,949,014	13,011,564	12,629,385	3,908,706	-
Strip Ratio		1.54		-	-	1.70	1.52	1.57	1.68	1.71	1.57	1.76	2.66	1.35	0.81	0.75	0.18	-
Operating Cost																		
Open Pit Mining	dollars	441,665,908	5.47	0	10,138,616	30,286,366	38,933,905	39,758,602	41,519,328	41,927,772	39,776,805	42,661,041	56,691,337	36,440,379	27,974,863	27,153,178	8,403,718	0
Processing	dollars	490,686,990	6.08	0	0	32,832,000	43,776,000	43,776,000	43,776,000	43,776,000	43,776,000	43,776,000	43,776,000	43,776,000	43,776,000	43,776,000	20,094,990	0
G&A	dollars	61,533,568	0.76	2,520,000	2,520,000	3,780,000	5,040,000	5,040,000	5,040,000	5,040,000	5,040,000	5,040,000	5,040,000	5,040,000	5,040,000	5,040,000	2,313,568	0
Subtotal Operating	dollars	993,886,465		2,520,000	12,658,616	66,898,366	87,749,905	88,574,602	90,335,328	90,743,772	88,592,805	91,477,041	105,507,337	85,256,379	76,790,863	75,969,178	30,812,276	0
Unit Cost																		
Operating Cost per tonne milled	$/tonne ore	$ 12.32		$ -	$ -	$ 12.39	$ 12.19	$ 12.30	$ 12.55	$ 12.60	$ 12.30	$ 12.71	$ 14.65	$ 11.84	$ 10.67	$ 10.55	$ 9.32	$ -
Mining Cost per Tonne Milled	$/tonne ore	$ 5.47		$ -	$ -	$ 5.61	$ 5.41	$ 5.52	$ 5.77	$ 5.82	$ 5.52	$ 5.93	$ 7.87	$ 5.06	$ 3.89	$ 3.77	$ 2.54	$ -
Capital Cost																		
Open Pit Mining	dollars	83,884,000		25,267,000	19,000,000	11,296,000	0	315,000	1,877,000	5,096,000	2,500,000	1,115,000	6,500,000	8,473,000	2,445,000	0	0	0
Underground Mining	dollars	0		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Processing	dollars	167,660,000		49,800,000	83,000,000	33,200,000	0	166,000	166,000	332,000	166,000	166,000	166,000	332,000	166,000	0	0	0
Infrastructure	dollars	140,900,000		23,800,000	68,655,000	4,355,000	13,090,000	2,051,061	2,051,061	2,051,061	2,055,050	7,380,872	4,650,000	3,100,000	3,100,000	3,100,000	1,460,896	0
Environment Costs	dollars	16,200,000		16,200,000	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Indirect	dollars	92,785,040		28,784,192	45,177,520	16,393,328	0	0	0	0	0	0	0	0	729,000	972,000	729,000	0
Contingency	dollars	61,296,600		20,176,500	29,433,300	9,256,800	0	0	0	0	0	0	0	0	729,000	972,000	729,000	0
Subtotal Capital	dollars	562,725,640		164,027,692	245,265,820	74,501,128	13,090,000	2,532,061	4,094,061	7,479,061	4,721,050	8,661,872	11,316,000	11,905,000	7,169,000	5,044,000	2,918,896	0
Revenue (after refining)																		
Gold Gross Revenue	dollars	1,459,000,693		-	-	79,056,727	108,689,069	144,681,350	171,828,962	116,484,750	144,089,350	155,770,749	129,924,931	115,375,519	129,474,377	115,182,917	48,441,993	-
less Gold Refining	dollars	12,286,322		-	-	665,741	915,276	1,218,369	1,446,981	980,924	1,213,384	1,311,754	1,094,105	971,583	1,090,311	969,961	407,933	-
less Gold Payables	dollars	21,885,010		-	-	1,185,851	1,630,336	2,170,220	2,577,434	1,747,271	2,161,340	2,336,561	1,948,874	1,730,633	1,942,116	1,727,744	726,630	-
Subtotal		1,493,172,025		-	-	77,205,135	106,143,457	141,292,761	167,804,547	113,756,555	140,714,626	152,122,434	126,881,952	112,673,303	126,441,951	112,485,211	47,307,430	-
less Royalty	dollars	14,248,294		-	-	772,051	1,061,435	1,412,928	1,678,045	1,137,566	1,407,146	1,521,224	1,268,820	1,126,733	1,264,420	1,124,852	473,074	-
Net Revenue	dollars	1,410,581,068		-	-	76,433,084	105,082,022	139,879,833	166,126,501	112,618,989	139,307,479	150,601,210	125,613,132	111,546,570	125,177,531	111,360,359	46,834,356	-
Cashflow																		
Operating Cost	dollars	993,886,000		2,520,000	12,659,000	66,898,000	87,750,000	88,575,000	90,335,000	90,744,000	88,593,000	91,477,000	105,507,000	85,256,000	76,791,000	75,969,000	30,812,000	0
Capital Cost	dollars	562,726,000		164,028,000	245,266,000	74,501,000	13,090,000	2,532,000	4,094,000	7,479,000	4,721,000	8,662,000	11,316,000	11,905,000	7,169,000	5,044,000	2,919,000	0
Revenue	dollars	1,410,581,000		0	0	76,433,000	105,082,000	139,880,000	166,127,000	112,619,000	139,307,000	150,601,000	125,613,000	111,547,000	125,178,000	111,360,000	46,834,000	0
Net Cashflow	dollars	-146,031,000		-166,548,000	-257,925,000	-64,966,000	4,242,000	48,773,000	71,698,000	14,396,000	45,993,000	50,462,000	8,790,000	14,386,000	41,218,000	30,347,000	13,103,000	0
Cumulative	dollars			-166,548,000	-424,473,000	-489,439,000	-485,197,000	-436,424,000	-364,726,000	-350,330,000	-304,337,000	-253,875,000	-245,085,000	-230,699,000	-189,481,000	-159,134,000	-146,031,000	0
NPV (millions) @	**0%**	-$146																
	5%	-$220																
	8%	-$245																
IRR		-5.0%																

Financial Base

Plant Throughput	**20,000**	**tonnes per day**																
		Total		**Year -2**	**Year -1**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	**Year 11**	**Year 12**	**Year 13**
Mill Production																		
Mill Feed	tonnes	80,705,097		-	-	5,400,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	3,305,097	-
Gold Grade	g/t	0.60		-	-	0.48	0.50	0.66	0.79	0.54	0.66	0.72	0.60	0.53	0.59	0.53	0.48	-
Gold Recovery	%			0.0%	0.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	0.0%
Insitu Gold	ounces	1,551,303		-	-	84,058	115,565	153,835	182,700	123,854	153,205	165,625	138,145	122,675	137,665	122,470	51,507	-
Recovered Gold	ounces	1,396,173		-	-	75,652	104,009	138,451	164,430	111,469	137,885	149,063	124,330	110,407	123,899	110,223	46,356	-
Mine Production																		
Open Pit																		
Ore																		
Ore to Mill	tonnes	80,028,004		-	-	4,722,907	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	3,305,097	-
Ore to Stockpile	tonnes	677,093		-	677,093	-	-	-	-	-	-	-	-	-	-	-	-	-
Stockpile to Mill	tonnes	677,093		-	-	677,093	-	-	-	-	-	-	-	-	-	-	-	-
Waste	tonnes	124,563,443		-	4,038,543	9,206,311	10,908,793	11,292,373	12,111,315	12,301,289	11,300,840	12,642,345	19,168,064	9,749,014	5,811,564	5,429,385	603,609	-
Total Material	tonnes	205,945,633		-	4,715,636	14,606,311	18,108,793	18,492,373	19,311,315	19,501,289	18,500,840	19,842,345	26,368,064	16,949,014	13,011,564	12,629,385	3,908,706	-
Strip Ratio		1.54		-	-	1.70	1.52	1.57	1.68	1.71	1.57	1.76	2.66	1.35	0.81	0.75	0.18	-
Operating Cost																		
Open Pit Mining	dollars	441,665,908	5.47	0	10,138,616	30,286,366	38,933,905	39,758,602	41,519,328	41,927,772	39,776,805	42,661,041	56,691,337	36,440,379	27,974,863	27,153,178	8,403,718	0
Processing	dollars	490,686,990	6.08	0	0	32,832,000	43,776,000	43,776,000	43,776,000	43,776,000	43,776,000	43,776,000	43,776,000	43,776,000	43,776,000	43,776,000	20,094,990	0
G&A	dollars	61,533,568	0.76	2,520,000	2,520,000	3,780,000	5,040,000	5,040,000	5,040,000	5,040,000	5,040,000	5,040,000	5,040,000	5,040,000	5,040,000	5,040,000	2,313,568	0
Subtotal Operating	dollars	993,886,465		2,520,000	12,658,616	66,898,366	87,749,905	88,574,602	90,335,328	90,743,772	88,592,805	91,477,041	105,507,337	85,256,379	76,790,863	75,969,178	30,812,276	0
Unit Cost																		
Operating Cost per tonne milled	$/tonne ore	$ 12.32		$ -	$ -	$ 12.39	$ 12.19	$ 12.30	$ 12.55	$ 12.60	$ 12.30	$ 12.71	$ 14.65	$ 11.84	$ 10.67	$ 10.55	$ 9.32	$ -
Mining Cost per Tonne Milled	$/tonne ore	$ 5.47		$ -	$ -	$ 5.61	$ 5.41	$ 5.52	$ 5.77	$ 5.82	$ 5.52	$ 5.93	$ 7.87	$ 5.06	$ 3.89	$ 3.77	$ 2.54	$ -
Capital Cost																		
Open Pit Mining	dollars	83,884,000		25,267,000	19,000,000	11,296,000	0	315,000	1,877,000	5,096,000	2,500,000	1,115,000	6,500,000	8,473,000	2,445,000	0	0	0
Underground Mining	dollars	0		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Processing	dollars	167,660,000		49,800,000	83,000,000	33,200,000	0	166,000	166,000	332,000	166,000	166,000	166,000	332,000	166,000	0	0	0
Infrastructure	dollars	140,900,000		23,800,000	68,655,000	4,355,000	13,090,000	2,051,061	2,051,061	2,051,061	2,055,050	7,380,872	4,650,000	3,100,000	3,100,000	3,100,000	1,460,896	0
Environment Costs	dollars	16,200,000		16,200,000	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Indirect	dollars	92,785,040		28,784,192	45,177,520	16,393,328	0	0	0	0	0	0	0	0	729,000	972,000	729,000	0
Contingency	dollars	61,296,600		20,176,500	29,433,300	9,256,800	0	0	0	0	0	0	0	0	729,000	972,000	729,000	0
Subtotal Capital	dollars	562,725,640		164,027,692	245,265,820	74,501,128	13,090,000	2,532,061	4,094,061	7,479,061	4,721,050	8,661,872	11,316,000	11,905,000	7,169,000	5,044,000	2,918,896	0
Revenue (after refining)																		
Gold Gross Revenue	dollars	1,866,599,413		-	-	101,142,680	139,053,363	185,100,751	219,832,548	149,026,911	184,343,364	199,288,178	166,221,853	147,607,796	165,645,429	147,361,386	61,975,156	-
less Gold Refining	dollars	11,504,465		-	-	623,376	857,032	1,140,837	1,354,900	918,502	1,136,169	1,228,278	1,024,480	909,755	1,020,927	908,237	381,973	-
less Gold Payables	dollars	27,998,991		-	-	1,517,140	2,085,800	2,776,511	3,297,488	2,235,404	2,765,150	2,989,323	2,493,328	2,214,117	2,484,681	2,210,421	929,627	-
Subtotal		1,906,102,869		-	-	99,002,164	136,110,531	181,183,403	215,180,159	145,873,006	180,442,045	195,070,576	162,704,046	144,483,923	162,139,821	144,242,729	60,663,555	-
less Royalty	dollars	18,270,960		-	-	990,022	1,361,105	1,811,834	2,151,802	1,458,730	1,804,420	1,950,706	1,627,040	1,444,839	1,621,398	1,442,427	606,636	-
Net Revenue	dollars	1,808,824,998		-	-	98,012,142	134,749,425	179,371,569	213,028,358	144,414,276	178,637,625	193,119,871	161,077,005	143,039,084	160,518,422	142,800,301	60,056,920	-
Cashflow																		
Operating Cost	dollars	993,886,000		2,520,000	12,659,000	66,898,000	87,750,000	88,575,000	90,335,000	90,744,000	88,593,000	91,477,000	105,507,000	85,256,000	76,791,000	75,969,000	30,812,000	0
Capital Cost	dollars	562,726,000		164,028,000	245,266,000	74,501,000	13,090,000	2,532,000	4,094,000	7,479,000	4,721,000	8,662,000	11,316,000	11,905,000	7,169,000	5,044,000	2,919,000	0
Revenue	dollars	1,808,824,000		0	0	98,012,000	134,749,000	179,372,000	213,028,000	144,414,000	178,638,000	193,120,000	161,077,000	143,039,000	160,518,000	142,800,000	60,057,000	0
Net Cashflow	dollars	252,212,000		-166,548,000	-257,925,000	-43,387,000	33,909,000	88,265,000	118,599,000	46,191,000	85,324,000	92,981,000	44,254,000	45,878,000	76,558,000	61,787,000	26,326,000	0
Cumulative	dollars			-166,548,000	-424,473,000	-467,860,000	-433,951,000	-345,686,000	-227,087,000	-180,896,000	-95,572,000	-2,591,000	41,663,000	87,541,000	164,099,000	225,886,000	252,212,000	0

NPV (millions) @	**0%**	$252
	5%	$49
	8%	-$28
IRR		6.8%

25K Cashflow

Engineering Base

Plant Throughput	**25,000**	**tonnes per day**															
		Total		**Year -2**	**Year -1**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	**Year 11**	**Year 12**
Mill Production																	
Mill Feed	tonnes	88,937,596		-	-	6,750,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	6,093,697	4,093,899	-
Gold Grade	g/t	0.57		-	-	0.51	0.51	0.73	0.64	0.64	0.65	0.53	0.51	0.53	0.55	0.43	-
Gold Recovery	%			0.0%	0.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	0.0%
Insitu Gold	ounces	1,642,985		-	-	110,219	146,300	210,218	184,165	185,136	188,311	153,843	146,624	153,202	108,034	56,933	-
Recovered Gold	ounces	1,478,686		-	-	99,197	131,670	189,197	165,748	166,622	169,480	138,459	131,962	137,881	97,230	51,240	-
Mine Production																	
Open Pit																	
Ore																	
Ore to Mill	tonnes	87,820,195		-	-	5,632,599	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	6,093,697	4,093,899	-
Ore to Stockpile	tonnes	1,117,401		-	1,117,401	-	-	-	-	-	-	-	-	-	-	-	-
Stockpile to Mill	tonnes	1,117,401		-	-	1,117,401	-	-	-	-	-	-	-	-	-	-	-
Waste	tonnes	129,429,262		-	4,766,910	9,711,382	11,882,769	11,137,134	11,643,620	14,599,589	16,033,360	16,159,364	15,479,628	9,901,105	6,384,894	1,729,508	-
Total Material	tonnes	219,484,259		-	5,884,311	16,461,382	20,882,769	20,137,134	20,643,620	23,599,589	25,033,360	25,159,364	24,479,628	18,901,105	12,478,591	5,823,407	-
Strip Ratio		1.46		-	-	1.44	1.32	1.24	1.29	1.62	1.78	1.80	1.72	1.10	1.05	0.42	-
Operating Cost																	
Open Pit Mining	dollars	454,761,765	5.11	0	12,239,367	32,474,181	43,436,158	41,885,239	42,938,729	49,087,145	52,069,388	52,331,478	50,917,626	39,314,297	25,955,469	12,112,687	0
Processing	dollars	518,506,185	5.83	0	0	39,352,500	52,470,000	52,470,000	52,470,000	52,470,000	52,470,000	52,470,000	52,470,000	52,470,000	35,526,254	23,867,431	0
G&A	dollars	57,783,182	0.65	2,655,000	2,655,000	3,982,500	5,310,000	5,310,000	5,310,000	5,310,000	5,310,000	5,310,000	5,310,000	5,310,000	3,595,281	2,415,400	0
Subtotal Operating	dollars	1,031,051,131		2,655,000	14,894,367	75,809,181	101,216,158	99,665,239	100,718,729	106,867,145	109,849,388	110,111,478	108,697,626	97,094,297	65,077,004	38,395,518	0
Unit Cost																	
Operating Cost per tonne milled	$/tonne ore	$ 11.59		$ -	$ -	$ 11.23	$ 11.25	$ 11.07	$ 11.19	$ 11.87	$ 12.21	$ 12.23	$ 12.08	$ 10.79	$ 10.68	$ 9.38	$ -
Mining Cost per Tonne Milled	$/tonne ore	$ 5.11		$ -	$ -	$ 4.81	$ 4.83	$ 4.65	$ 4.77	$ 5.45	$ 5.79	$ 5.81	$ 5.66	$ 4.37	$ 4.26	$ 2.96	$ -
Capital Cost																	
Open Pit Mining	dollars	79,939,000		28,467,000	20,500,000	9,796,000	0	315,000	5,077,000	9,796,000	1,000,000	1,115,000	3,300,000	573,000	0	0	0
Underground Mining	dollars	0		0	0	0	0	0	0	0	0	0	0	0	0	0	0
Processing	dollars	193,819,000		57,570,000	95,950,000	38,380,000	0	191,900	191,900	383,800	191,900	383,800	191,900	191,900	191,900	0	0
Infrastructure	dollars	140,900,000		23,800,000	68,655,000	4,355,000	13,090,000	2,790,000	2,790,000	2,480,000	2,480,000	7,380,872	4,650,000	2,790,000	2,790,000	2,849,128	0
Environment Costs	dollars	16,200,000		16,200,000	0	0	0	0	0	0	0	0	0	0	0	0	0
Indirect	dollars	100,342,876		30,972,643	48,956,438	17,983,795	0	0	0	0	0	0	0	729,000	972,000	729,000	0
Contingency	dollars	64,628,700		21,057,780	31,099,350	10,041,570	0	0	0	0	0	0	0	729,000	972,000	729,000	0
Subtotal Capital	dollars	595,829,576		178,067,423	265,160,788	80,556,365	13,090,000	3,296,900	8,058,900	12,659,800	3,671,900	8,879,672	8,141,900	5,012,900	4,925,900	4,307,128	0
Revenue (after refining)																	
Gold Gross Revenue	dollars	1,545,227,116		-	-	103,660,871	137,595,232	197,710,379	173,206,837	174,120,399	177,106,130	144,689,208	137,900,186	144,086,104	101,605,864	53,545,908	-
less Gold Refining	dollars	13,012,439		-	-	872,934	1,158,697	1,664,930	1,458,584	1,466,277	1,491,420	1,218,435	1,161,265	1,213,357	855,628	450,913	-
less Gold Payables	dollars	23,178,407		-	-	1,554,913	2,063,928	2,965,656	2,598,103	2,611,806	2,656,592	2,170,338	2,068,503	2,161,292	1,524,088	803,189	-
Subtotal		1,581,417,962		-	-	101,233,024	134,372,607	193,079,793	169,150,151	170,042,316	172,958,118	141,300,434	134,670,419	140,711,456	99,226,147	52,291,806	-
less Royalty	dollars	15,090,363		-	-	1,012,330	1,343,726	1,930,798	1,691,502	1,700,423	1,729,581	1,413,004	1,346,704	1,407,115	992,261	522,918	-
Net Revenue	dollars	1,493,945,908		-	-	100,220,694	133,028,881	191,148,996	167,458,649	168,341,893	171,228,536	139,887,430	133,323,714	139,304,341	98,233,886	51,768,888	-
Cashflow																	
Operating Cost	dollars	1,031,050,000		2,655,000	14,894,000	75,809,000	101,216,000	99,665,000	100,719,000	106,867,000	109,849,000	110,111,000	108,698,000	97,094,000	65,077,000	38,396,000	0
Capital Cost	dollars	595,830,000		178,067,000	265,161,000	80,556,000	13,090,000	3,297,000	8,059,000	12,660,000	3,672,000	8,880,000	8,142,000	5,013,000	4,926,000	4,307,000	0
Revenue	dollars	1,493,947,000		0	0	100,221,000	133,029,000	191,149,000	167,459,000	168,342,000	171,229,000	139,887,000	133,324,000	139,304,000	98,234,000	51,769,000	0
Net Cashflow	dollars	-132,933,000		-180,722,000	-280,055,000	-56,144,000	18,723,000	88,187,000	58,681,000	48,815,000	57,708,000	20,896,000	16,484,000	37,197,000	28,231,000	9,066,000	0
Cumulative	dollars			-180,722,000	-460,777,000	-516,921,000	-498,198,000	-410,011,000	-351,330,000	-302,515,000	-244,807,000	-223,911,000	-207,427,000	-170,230,000	-141,999,000	-132,933,000	0

NPV (millions) @	**0%**	-$133
	5%	-$207
	8%	-$233
IRR		-4.9%

Financial Base

Plant Throughput	25,000	tonnes per day															
		Total		Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Mill Production																	
Mill Feed	tonnes	88,937,596		-	-	6,750,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	6,093,697	4,093,899	-
Gold Grade	g/t	0.57		-	-	0.51	0.51	0.73	0.64	0.64	0.65	0.53	0.51	0.53	0.55	0.43	-
Gold Recovery	%			0.0%	0.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	0.0%
Insitu Gold	ounces	1,642,985		-	-	110,219	146,300	210,218	184,165	185,136	188,311	153,843	146,624	153,202	108,034	56,933	-
Recovered Gold	ounces	1,478,686		-	-	99,197	131,670	189,197	165,748	166,622	169,480	138,459	131,962	137,881	97,230	51,240	-
Mine Production																	
Open Pit																	
Ore																	
Ore to Mill	tonnes	87,820,195		-	-	5,632,599	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	6,093,697	4,093,899	-
Ore to Stockpile	tonnes	1,117,401		-	1,117,401	-	-	-	-	-	-	-	-	-	-	-	-
Stockpile to Mill	tonnes	1,117,401		-	-	1,117,401	-	-	-	-	-	-	-	-	-	-	-
Waste	tonnes	129,429,262		-	4,766,910	9,711,382	11,882,769	11,137,134	11,643,620	14,599,589	16,033,360	16,159,364	15,479,628	9,901,105	6,384,894	1,729,508	-
Total Material	tonnes	219,484,259		-	5,884,311	16,461,382	20,882,769	20,137,134	20,643,620	23,599,589	25,033,360	25,159,364	24,479,628	18,901,105	12,478,591	5,823,407	-
Strip Ratio		1.46		-	-	1.44	1.32	1.24	1.29	1.62	1.78	1.80	1.72	1.10	1.05	0.42	-
Operating Cost																	
Open Pit Mining	dollars	454,761,765	5.11	0	12,239,367	32,474,181	43,436,158	41,885,239	42,938,729	49,087,145	52,069,388	52,331,478	50,917,626	39,314,297	25,955,469	12,112,687	0
Processing	dollars	518,506,185	5.83	0	0	39,352,500	52,470,000	52,470,000	52,470,000	52,470,000	52,470,000	52,470,000	52,470,000	52,470,000	35,526,254	23,867,431	0
G&A	dollars	57,783,182	0.65	2,655,000	2,655,000	3,982,500	5,310,000	5,310,000	5,310,000	5,310,000	5,310,000	5,310,000	5,310,000	5,310,000	3,595,281	2,415,400	0
Subtotal Operating	dollars	1,031,051,131		2,655,000	14,894,367	75,809,181	101,216,158	99,665,239	100,718,729	106,867,145	109,849,388	110,111,478	108,697,626	97,094,297	65,077,004	38,395,518	0
Unit Cost																	
Operating Cost per tonne milled	$/tonne ore	$ 11.59		$ -	$ -	$ 11.23	$ 11.25	$ 11.07	$ 11.19	$ 11.87	$ 12.21	$ 12.23	$ 12.08	$ 10.79	$ 10.68	$ 9.38	$ -
Mining Cost per Tonne Milled	$/tonne ore	$ 5.11		$ -	$ -	$ 4.81	$ 4.83	$ 4.65	$ 4.77	$ 5.45	$ 5.79	$ 5.81	$ 5.66	$ 4.37	$ 4.26	$ 2.96	$ -
Capital Cost																	
Open Pit Mining	dollars	79,939,000		28,467,000	20,500,000	9,796,000	0	315,000	5,077,000	9,796,000	1,000,000	1,115,000	3,300,000	573,000	0	0	0
Underground Mining	dollars	0		0	0	0	0	0	0	0	0	0	0	0	0	0	0
Processing	dollars	193,819,000		57,570,000	95,950,000	38,380,000	0	191,900	191,900	383,800	191,900	383,800	191,900	191,900	191,900	0	0
Infrastructure	dollars	140,900,000		23,800,000	68,655,000	4,355,000	13,090,000	2,790,000	2,790,000	2,480,000	2,480,000	7,380,872	4,650,000	2,790,000	2,790,000	2,849,128	0
Environment Costs	dollars	16,200,000		16,200,000	0	0	0	0	0	0	0	0	0	0	0	0	0
Indirect	dollars	100,342,876		30,972,643	48,956,438	17,983,795	0	0	0	0	0	0	0	729,000	972,000	729,000	0
Contingency	dollars	64,628,700		21,057,780	31,099,350	10,041,570	0	0	0	0	0	0	0	729,000	972,000	729,000	0
Subtotal Capital	dollars	595,829,576		178,067,423	265,160,788	80,556,365	13,090,000	3,296,900	8,058,900	12,659,800	3,671,900	8,879,672	8,141,900	5,012,900	4,925,900	4,307,128	0
Revenue (after refining)																	
Gold Gross Revenue	dollars	1,976,914,775		-	-	132,620,444	176,034,995	252,944,415	221,595,358	222,764,140	226,583,989	185,110,803	176,425,143	184,339,211	129,991,333	68,504,943	-
less Gold Refining	dollars	12,184,375		-	-	817,383	1,084,961	1,558,979	1,365,765	1,372,969	1,396,511	1,140,899	1,087,366	1,136,143	801,179	422,218	-
less Gold Payables	dollars	29,653,722		-	-	1,989,307	2,640,525	3,794,166	3,323,930	3,341,462	3,398,760	2,776,662	2,646,377	2,765,088	1,949,870	1,027,574	-
Subtotal		2,018,752,872		-	-	129,813,754	172,309,508	247,591,269	216,905,662	218,049,709	221,788,718	181,193,243	172,691,400	180,437,980	127,240,284	67,055,151	-
less Royalty	dollars	19,350,767		-	-	1,298,138	1,723,095	2,475,913	2,169,057	2,180,497	2,217,887	1,811,932	1,726,914	1,804,380	1,272,403	670,552	-
Net Revenue	dollars	1,915,725,912		-	-	128,515,617	170,586,413	245,115,357	214,736,606	215,869,212	219,570,831	179,381,310	170,964,486	178,633,600	125,967,881	66,384,599	-
Cashflow																	
Operating Cost	dollars	1,031,050,000		2,655,000	14,894,000	75,809,000	101,216,000	99,665,000	100,719,000	106,867,000	109,849,000	110,111,000	108,698,000	97,094,000	65,077,000	38,396,000	0
Capital Cost	dollars	595,830,000		178,067,000	265,161,000	80,556,000	13,090,000	3,297,000	8,059,000	12,660,000	3,672,000	8,880,000	8,142,000	5,013,000	4,926,000	4,307,000	0
Revenue	dollars	1,915,726,000		0	0	128,516,000	170,586,000	245,115,000	214,737,000	215,869,000	219,571,000	179,381,000	170,964,000	178,634,000	125,968,000	66,385,000	0
Net Cashflow	dollars	288,846,000		-180,722,000	-280,055,000	-27,849,000	56,280,000	142,153,000	105,959,000	96,342,000	106,050,000	60,390,000	54,124,000	76,527,000	55,965,000	23,682,000	0
Cumulative	dollars			-180,722,000	-460,777,000	-488,626,000	-432,346,000	-290,193,000	-184,234,000	-87,892,000	18,158,000	78,548,000	132,672,000	209,199,000	265,164,000	288,846,000	0

NPV (millions) @	**0%**	$289
	5%	$87
	8%	$8
IRR		8.4%

30K Cashflow

Engineering Base

Plant Throughput	**30,000**	**tonnes per day**															
		Total		**Year -2**	**Year -1**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	**Year 11**	**Year 12**
Mill Production																	
Mill Feed	tonnes	97,091,098		-	-	8,100,000	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	7,964,886	5,426,212	-	-
Gold Grade	g/t	0.56		-	-	0.46	0.48	0.71	0.65	0.65	0.56	0.47	0.51	0.57	0.43	-	-
Gold Recovery	%			0.0%	0.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	0.0%	0.0%
Insitu Gold	ounces	1,740,964		-	-	118,544	168,301	245,568	225,739	226,826	195,677	164,040	177,060	144,808	74,401	-	-
Recovered Gold	ounces	1,566,867		-	-	106,690	151,471	221,011	203,165	204,143	176,110	147,636	159,354	130,327	66,961	-	-
Mine Production																	
Open Pit																	
Ore																	
Ore to Mill	tonnes	95,149,516		-	-	6,158,418	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	7,964,886	5,426,212	-	-
Ore to Stockpile	tonnes	1,941,582		-	1,941,582	-	-	-	-	-	-	-	-	-	-	-	-
Stockpile to Mill	tonnes	1,941,582		-	-	1,941,582	-	-	-	-	-	-	-	-	-	-	-
Waste	tonnes	141,213,379		-	6,187,670	10,655,344	11,118,593	16,912,609	16,577,516	16,287,524	19,744,515	19,733,798	13,284,211	7,837,425	2,874,175	-	-
Total Material	tonnes	240,246,059		-	8,129,252	18,755,344	21,918,593	27,712,609	27,377,516	27,087,524	30,544,515	30,533,798	24,084,211	15,802,311	8,300,387	-	-
Strip Ratio		1.45		-	-	1.32	1.03	1.57	1.53	1.51	1.83	1.83	1.23	0.98	0.53	-	-
Operating Cost																	
Open Pit Mining	dollars	477,579,745	4.92	0	16,258,504	34,598,315	43,837,186	55,425,218	54,755,032	54,175,047	61,089,030	61,067,596	48,168,422	31,604,622	16,600,774	0	0
Processing	dollars	549,535,615	5.66	0	0	45,846,000	61,128,000	61,128,000	61,128,000	61,128,000	61,128,000	61,128,000	61,128,000	45,081,255	30,712,360	0	0
G&A	dollars	60,419,015	0.62	3,024,000	3,024,000	4,536,000	6,048,000	6,048,000	6,048,000	6,048,000	6,048,000	6,048,000	6,048,000	4,460,336	3,038,679	0	0
Subtotal Operating	dollars	1,087,534,375		3,024,000	19,282,504	84,980,315	111,013,186	122,601,218	121,931,032	121,351,047	128,265,030	128,243,596	115,344,422	81,146,213	50,351,813	0	0
Unit Cost																	
Operating Cost per tonne milled	$/tonne ore	$ 11.20		$ -	$ -	$ 10.49	$ 10.28	$ 11.35	$ 11.29	$ 11.24	$ 11.88	$ 11.87	$ 10.68	$ 10.19	$ 9.28	$ -	$ -
Mining Cost per Tonne Milled	$/tonne ore	$ 4.92		$ -	$ -	$ 4.27	$ 4.06	$ 5.13	$ 5.07	$ 5.02	$ 5.66	$ 5.65	$ 4.46	$ 3.97	$ 3.06	$ -	$ -
Capital Cost																	
Open Pit Mining	dollars	95,166,000		30,667,000	21,500,000	6,596,000	15,800,000	315,000	1,877,000	12,996,000	4,300,000	1,115,000	0	0	0	0	0
Underground Mining	dollars	0		0	0	0	0	0	0	0	0	0	0	0	0	0	0
Processing	dollars	218,261,000		64,830,000	108,050,000	43,220,000	0	432,200	216,100	432,200	216,100	216,100	432,200	216,100	0	0	0
Infrastructure	dollars	140,900,000		23,800,000	68,655,000	4,355,000	13,090,000	2,790,000	3,100,000	3,100,000	4,650,000	7,380,872	2,790,000	3,100,000	4,089,128	0	0
Environment Costs	dollars	16,200,000		16,200,000	0	0	0	0	0	0	0	0	0	0	0	0	0
Indirect	dollars	109,295,944		33,963,103	53,432,972	19,469,869	0	0	0	0	0	0	729,000	972,000	729,000	0	0
Contingency	dollars	70,579,050		23,299,695	34,074,525	10,774,830	0	0	0	0	0	0	729,000	972,000	729,000	0	0
Subtotal Capital	dollars	650,401,994		192,759,798	285,712,497	84,415,699	28,890,000	3,537,200	5,193,100	16,528,200	9,166,100	8,711,972	4,680,200	5,260,100	5,547,128	0	0
Revenue (after refining)																	
Gold Gross Revenue	dollars	1,637,376,508		-	-	111,490,610	158,287,414	230,956,968	212,307,100	213,329,918	184,034,617	154,279,505	166,524,996	136,191,484	69,973,895	-	-
less Gold Refining	dollars	13,788,434		-	-	938,868	1,332,947	1,944,901	1,787,849	1,796,462	1,549,765	1,299,196	1,402,316	1,146,876	589,254	-	-
less Gold Payables	dollars	24,560,648		-	-	1,672,359	2,374,311	3,464,355	3,184,607	3,199,949	2,760,519	2,314,193	2,497,875	2,042,872	1,049,608	-	-
Subtotal		1,675,725,589		-	-	108,879,382	154,580,156	225,547,713	207,334,644	208,333,507	179,724,333	150,666,117	162,624,805	133,001,736	68,335,033	-	-
less Royalty	dollars	15,990,274		-	-	1,088,794	1,545,802	2,255,477	2,073,346	2,083,335	1,797,243	1,506,661	1,626,248	1,330,017	683,350	-	-
Net Revenue	dollars	1,583,037,152		-	-	107,790,588	153,034,355	223,292,236	205,261,298	206,250,171	177,927,090	149,159,456	160,998,557	131,671,719	67,651,683	-	-
Cashflow																	
Operating Cost	dollars	1,087,534,000		3,024,000	19,283,000	84,980,000	111,013,000	122,601,000	121,931,000	121,351,000	128,265,000	128,244,000	115,344,000	81,146,000	50,352,000	0	0
Capital Cost	dollars	650,401,000		192,760,000	285,712,000	84,416,000	28,890,000	3,537,000	5,193,000	16,528,000	9,166,000	8,712,000	4,680,000	5,260,000	5,547,000	0	0
Revenue	dollars	1,583,037,000		0	0	107,791,000	153,034,000	223,292,000	205,261,000	206,250,000	177,927,000	149,159,000	160,999,000	131,672,000	67,652,000	0	0
Net Cashflow	dollars	-154,898,000		-195,784,000	-304,995,000	-61,605,000	13,131,000	97,154,000	78,137,000	68,371,000	40,496,000	12,203,000	40,975,000	45,266,000	11,753,000	0	0
Cumulative	dollars			-195,784,000	-500,779,000	-562,384,000	-549,253,000	-452,099,000	-373,962,000	-305,591,000	-265,095,000	-252,892,000	-211,917,000	-166,651,000	-154,898,000	0	0

NPV (millions) @	**0%**	-$155
	5%	-$229
	8%	-$256
IRR		-5.5%

Financial Base

Plant Throughput **30,000** **tonnes per day**

		Total		Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Mill Production																	
Mill Feed	tonnes	97,091,098		-	-	8,100,000	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	7,964,886	5,426,212	-	-
Gold Grade	g/t	0.56		-	-	0.46	0.48	0.71	0.65	0.65	0.56	0.47	0.51	0.57	0.43	-	-
Gold Recovery	%			0.0%	0.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	0.0%	0.0%
Insitu Gold	ounces	1,740,964		-	-	118,544	168,301	245,568	225,739	226,826	195,677	164,040	177,060	144,808	74,401	-	-
Recovered Gold	ounces	1,566,867		-	-	106,690	151,471	221,011	203,165	204,143	176,110	147,636	159,354	130,327	66,961	-	-
Mine Production																	
Open Pit																	
Ore																	
Ore to Mill	tonnes	95,149,516		-	-	6,158,418	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	7,964,886	5,426,212	-	-
Ore to Stockpile	tonnes	1,941,582		-	1,941,582	-	-	-	-	-	-	-	-	-	-	-	-
Stockpile to Mill	tonnes	1,941,582		-	-	1,941,582	-	-	-	-	-	-	-	-	-	-	-
Waste	tonnes	141,213,379		-	6,187,670	10,655,344	11,118,593	16,912,609	16,577,516	16,287,524	19,744,515	19,733,798	13,284,211	7,837,425	2,874,175	-	-
Total Material	tonnes	240,246,059		-	8,129,252	18,755,344	21,918,593	27,712,609	27,377,516	27,087,524	30,544,515	30,533,798	24,084,211	15,802,311	8,300,387	-	-
Strip Ratio		1.45		-	-	1.32	1.03	1.57	1.53	1.51	1.83	1.83	1.23	0.98	0.53	-	-
Operating Cost																	
Open Pit Mining	dollars	477,579,745	4.92	0	16,258,504	34,598,315	43,837,186	55,425,218	54,755,032	54,175,047	61,089,030	61,067,596	48,168,422	31,604,622	16,600,774	0	0
Processing	dollars	549,535,615	5.66	0	0	45,846,000	61,128,000	61,128,000	61,128,000	61,128,000	61,128,000	61,128,000	61,128,000	45,081,255	30,712,360	0	0
G&A	dollars	60,419,015	0.62	3,024,000	3,024,000	4,536,000	6,048,000	6,048,000	6,048,000	6,048,000	6,048,000	6,048,000	6,048,000	4,460,336	3,038,679	0	0
Subtotal Operating	dollars	1,087,534,375		3,024,000	19,282,504	84,980,315	111,013,186	122,601,218	121,931,032	121,351,047	128,265,030	128,243,596	115,344,422	81,146,213	50,351,813	0	0
Unit Cost																	
Operating Cost per tonne milled	$/tonne ore	$ 11.20		$ -	$ -	$ 10.49	$ 10.28	$ 11.35	$ 11.29	$ 11.24	$ 11.88	$ 11.87	$ 10.68	$ 10.19	$ 9.28	$ -	$ -
Mining Cost per Tonne Milled	$/tonne ore	$ 4.92		$ -	$ -	$ 4.27	$ 4.06	$ 5.13	$ 5.07	$ 5.02	$ 5.66	$ 5.65	$ 4.46	$ 3.97	$ 3.06	$ -	$ -
Capital Cost																	
Open Pit Mining	dollars	95,166,000		30,667,000	21,500,000	6,596,000	15,800,000	315,000	1,877,000	12,996,000	4,300,000	1,115,000	0	0	0	0	0
Underground Mining	dollars	0		0	0	0	0	0	0	0	0	0	0	0	0	0	0
Processing	dollars	218,261,000		64,830,000	108,050,000	43,220,000	0	432,200	216,100	432,200	216,100	216,100	432,200	216,100	0	0	0
Infrastructure	dollars	140,900,000		23,800,000	68,655,000	4,355,000	13,090,000	2,790,000	3,100,000	3,100,000	4,650,000	7,380,872	2,790,000	3,100,000	4,089,128	0	0
Environment Costs	dollars	16,200,000		16,200,000	0	0	0	0	0	0	0	0	0	0	0	0	0
Indirect	dollars	109,295,944		33,963,103	53,432,972	19,469,869	0	0	0	0	0	0	729,000	972,000	729,000	0	0
Contingency	dollars	70,579,050		23,299,695	34,074,525	10,774,830	0	0	0	0	0	0	729,000	972,000	729,000	0	0
Subtotal Capital	dollars	650,401,994		192,759,798	285,712,497	84,415,699	28,890,000	3,537,200	5,193,100	16,528,200	9,166,100	8,711,972	4,680,200	5,260,100	5,547,128	0	0
Revenue (after refining)																	
Gold Gross Revenue	dollars	2,094,807,797		-	-	142,637,565	202,507,919	295,479,052	271,618,999	272,927,560	235,448,078	197,380,327	213,046,821	174,239,084	89,522,392	-	-
less Gold Refining	dollars	12,910,988		-	-	879,122	1,248,123	1,821,134	1,674,077	1,682,142	1,451,144	1,216,520	1,313,077	1,073,893	551,756	-	-
less Gold Payables	dollars	31,422,117		-	-	2,139,563	3,037,619	4,432,186	4,074,285	4,093,913	3,531,721	2,960,705	3,195,702	2,613,586	1,342,836	-	-
Subtotal		2,139,140,902		-	-	139,618,880	198,222,177	289,225,731	265,870,637	267,151,505	230,465,213	193,203,102	208,538,041	170,551,605	87,627,800	-	-
less Royalty	dollars	20,504,747		-	-	1,396,189	1,982,222	2,892,257	2,658,706	2,671,515	2,304,652	1,932,031	2,085,380	1,705,516	876,278	-	-
Net Revenue	dollars	2,029,969,946		-	-	138,222,691	196,239,956	286,333,474	263,211,931	264,479,989	228,160,561	191,271,071	206,452,661	168,846,089	86,751,522	-	-
Cashflow																	
Operating Cost	dollars	1,087,534,000		3,024,000	19,283,000	84,980,000	111,013,000	122,601,000	121,931,000	121,351,000	128,265,000	128,244,000	115,344,000	81,146,000	50,352,000	0	0
Capital Cost	dollars	650,401,000		192,760,000	285,712,000	84,416,000	28,890,000	3,537,000	5,193,000	16,528,000	9,166,000	8,712,000	4,680,000	5,260,000	5,547,000	0	0
Revenue	dollars	2,029,971,000		0	0	138,223,000	196,240,000	286,333,000	263,212,000	264,480,000	228,161,000	191,271,000	206,453,000	168,846,000	86,752,000	0	0
Net Cashflow	dollars	292,036,000		-195,784,000	-304,995,000	-31,173,000	56,337,000	160,195,000	136,088,000	126,601,000	90,730,000	54,315,000	86,429,000	82,440,000	30,853,000	0	0
Cumulative	dollars			-195,784,000	-500,779,000	-531,952,000	-475,615,000	-315,420,000	-179,332,000	-52,731,000	37,999,000	92,314,000	178,743,000	261,183,000	292,036,000	0	0

NPV (millions) @	**0%**	$292
	5%	$88
	8%	$6
IRR		8.3%

35K Cashflow

Engineering Base

Plant Throughput	**35,000**	**tonnes per day**														
		Total		**Year -2**	**Year -1**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	**Year 11**	**Year 12**
Mill Production																
Mill Feed	tonnes	121,779,879		-	-	9,450,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	7,582,974	3,946,905	-
Gold Grade	g/t	0.53		-	-	0.50	0.55	0.67	0.54	0.59	0.49	0.47	0.49	0.52	0.47	0.62	-
Gold Recovery	%			0.0%	0.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	0.0%
Insitu Gold	ounces	2,091,670		-	-	150,990	221,846	270,108	218,812	239,213	199,143	190,946	198,293	210,500	113,505	78,314	-
Recovered Gold	ounces	1,882,503		-	-	135,891	199,661	243,097	196,931	215,292	179,228	171,851	178,463	189,450	102,155	70,483	-
Mine Production																
Open Pit																
Ore																
Ore to Mill	tonnes	119,001,816		-	-	6,671,937	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	7,582,974	3,946,905	-
Ore to Stockpile	tonnes	2,778,063		131,164	2,646,899	-	-	-	-	-	-	-	-	-	-	-	-
Stockpile to Mill	tonnes	2,778,063		-	-	2,778,063	-	-	-	-	-	-	-	-	-	-	-
Waste	tonnes	207,062,345		1,259,831	6,976,121	11,379,032	16,381,333	16,479,780	24,646,409	25,855,792	25,905,419	25,074,704	17,761,322	23,935,516	11,072,445	334,643	-
Total Material	tonnes	331,620,287		1,390,995	9,623,020	20,829,032	28,981,333	29,079,780	37,246,409	38,455,792	38,505,419	37,674,704	30,361,322	36,535,516	18,655,419	4,281,548	-
Strip Ratio		1.70		-	-	1.20	1.30	1.31	1.96	2.05	2.06	1.99	1.41	1.90	1.46	0.08	-
Operating Cost																
Open Pit Mining	dollars	639,342,946	5.25	2,698,530	18,668,659	36,407,911	56,223,785	56,414,772	72,258,033	74,604,236	74,700,512	73,088,926	58,900,964	70,878,901	36,191,512	8,306,203	0
Processing	dollars	669,789,335	5.50	0	0	51,975,000	69,300,000	69,300,000	69,300,000	69,300,000	69,300,000	69,300,000	69,300,000	69,300,000	41,706,357	21,707,978	0
G&A	dollars	72,565,135	0.60	3,402,000	3,402,000	5,103,000	6,804,000	6,804,000	6,804,000	6,804,000	6,804,000	6,804,000	6,804,000	6,804,000	4,094,806	2,131,329	0
Subtotal Operating	dollars	1,381,697,415		6,100,530	22,070,659	93,485,911	132,327,785	132,518,772	148,362,033	150,708,236	150,804,512	149,192,926	135,004,964	146,982,901	81,992,675	32,145,509	0
Unit Cost																
Operating Cost per tonne milled	$/tonne ore	$ 11.35		$ -	$ -	$ 9.89	$ 10.50	$ 10.52	$ 11.77	$ 11.96	$ 11.97	$ 11.84	$ 10.71	$ 11.67	$ 10.81	$ 8.14	$ -
Mining Cost per Tonne Milled	$/tonne ore	$ 5.25		$ -	$ -	$ 3.85	$ 4.46	$ 4.48	$ 5.73	$ 5.92	$ 5.93	$ 5.80	$ 4.67	$ 5.63	$ 4.77	$ 2.10	$ -
Capital Cost																
Open Pit Mining	dollars	118,539,000		36,867,000	18,500,000	23,196,000	0	20,815,000	5,077,000	7,396,000	1,000,000	315,000	4,800,000	573,000	0	0	0
Underground Mining	dollars	0		0	0	0	0	0	0	0	0	0	0	0	0	0	0
Processing	dollars	241,188,000		71,640,000	119,400,000	47,760,000	0	238,800	238,800	477,600	238,800	477,600	238,800	238,800	238,800	0	0
Infrastructure	dollars	140,900,000		23,800,000	68,655,000	4,355,000	13,090,000	3,100,000	3,100,000	3,100,000	3,100,000	7,440,000	3,100,000	3,100,000	3,100,000	1,860,000	0
Environment Costs	dollars	16,200,000		16,200,000	0	0	0	0	0	0	0	0	0	0	0	0	0
Indirect	dollars	118,603,052		37,222,696	58,086,526	20,863,830	0	0	0	0	0	0	0	729,000	972,000	729,000	0
Contingency	dollars	77,524,050		26,084,385	37,547,025	11,462,640	0	0	0	0	0	0	0	729,000	972,000	729,000	0
Subtotal Capital	dollars	712,954,102		211,814,081	302,188,551	107,637,470	13,090,000	24,153,800	8,415,800	10,973,600	4,338,800	8,232,600	8,138,800	5,369,800	5,282,800	3,318,000	0
Revenue (after refining)																
Gold Gross Revenue	dollars	1,967,215,431		-	-	142,006,342	208,646,108	254,036,344	205,792,754	224,979,881	187,293,748	179,584,798	186,494,236	197,975,082	106,751,828	73,654,310	-
less Gold Refining	dollars	16,566,025		-	-	1,195,843	1,757,020	2,139,253	1,732,992	1,894,567	1,577,211	1,512,293	1,570,478	1,667,159	898,963	620,247	-
less Gold Payables	dollars	29,508,231		-	-	2,130,095	3,129,692	3,810,545	3,086,891	3,374,698	2,809,406	2,693,772	2,797,414	2,969,626	1,601,277	1,104,815	-
Subtotal		2,013,289,687		-	-	138,680,404	203,759,396	248,086,546	200,972,871	219,710,615	182,907,131	175,378,733	182,126,345	193,338,298	104,251,588	71,929,248	-
less Royalty	dollars	19,211,412		-	-	1,386,804	2,037,594	2,480,865	2,009,729	2,197,106	1,829,071	1,753,787	1,821,263	1,933,383	1,042,516	719,292	-
Net Revenue	dollars	1,901,929,763		-	-	137,293,600	201,721,802	245,605,680	198,963,142	217,513,509	181,078,060	173,624,946	180,305,082	191,404,915	103,209,072	71,209,956	-
Cashflow																
Operating Cost	dollars	1,381,700,000		6,101,000	22,071,000	93,486,000	132,328,000	132,519,000	148,362,000	150,708,000	150,805,000	149,193,000	135,005,000	146,983,000	81,993,000	32,146,000	0
Capital Cost	dollars	712,956,000		211,814,000	302,189,000	107,637,000	13,090,000	24,154,000	8,416,000	10,974,000	4,339,000	8,233,000	8,139,000	5,370,000	5,283,000	3,318,000	0
Revenue	dollars	1,901,931,000		0	0	137,294,000	201,722,000	245,606,000	198,963,000	217,514,000	181,078,000	173,625,000	180,305,000	191,405,000	103,209,000	71,210,000	0
Net Cashflow	dollars	-192,725,000		-217,915,000	-324,260,000	-63,829,000	56,304,000	88,933,000	42,185,000	55,832,000	25,934,000	16,199,000	37,161,000	39,052,000	15,933,000	35,746,000	0
Cumulative	dollars			-217,915,000	-542,175,000	-606,004,000	-549,700,000	-460,767,000	-418,582,000	-362,750,000	-336,816,000	-320,617,000	-283,456,000	-244,404,000	-228,471,000	-192,725,000	0

NPV (millions) @	**0%**	-$193
	5%	-$268
	8%	-$294
IRR		-6.1%

Financial Base

Plant Throughput	35,000	tonnes per day															
		Total		Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Mill Production																	
Mill Feed	tonnes	121,779,879		-	-	9,450,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	7,582,974	3,946,905	-
Gold Grade	g/t	0.53		-	-	0.50	0.55	0.67	0.54	0.59	0.49	0.47	0.49	0.52	0.47	0.62	-
Gold Recovery	%			0.0%	0.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	0.0%
Insitu Gold	ounces	2,091,670		-	-	150,990	221,846	270,108	218,812	239,213	199,143	190,946	198,293	210,500	113,505	78,314	-
Recovered Gold	ounces	1,882,503		-	-	135,891	199,661	243,097	196,931	215,292	179,228	171,851	178,463	189,450	102,155	70,483	-
Mine Production																	
Open Pit																	
Ore																	
Ore to Mill	tonnes	119,001,816		-	-	6,671,937	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	7,582,974	3,946,905	-
Ore to Stockpile	tonnes	2,778,063		131,164	2,646,899	-	-	-	-	-	-	-	-	-	-	-	-
Stockpile to Mill	tonnes	2,778,063		-	-	2,778,063	-	-	-	-	-	-	-	-	-	-	-
Waste	tonnes	207,062,345		1,259,831	6,976,121	11,379,032	16,381,333	16,479,780	24,646,409	25,855,792	25,905,419	25,074,704	17,761,322	23,935,516	11,072,445	334,643	-
Total Material	tonnes	331,620,287		1,390,995	9,623,020	20,829,032	28,981,333	29,079,780	37,246,409	38,455,792	38,505,419	37,674,704	30,361,322	36,535,516	18,655,419	4,281,548	-
Strip Ratio		1.70		-	-	1.20	1.30	1.31	1.96	2.05	2.06	1.99	1.41	1.90	1.46	0.08	-
Operating Cost																	
Open Pit Mining	dollars	639,342,946	5.25	2,698,530	18,668,659	36,407,911	56,223,785	56,414,772	72,258,033	74,604,236	74,700,512	73,088,926	58,900,964	70,878,901	36,191,512	8,306,203	0
Processing	dollars	669,789,335	5.50	0	0	51,975,000	69,300,000	69,300,000	69,300,000	69,300,000	69,300,000	69,300,000	69,300,000	69,300,000	41,706,357	21,707,978	0
G&A	dollars	72,565,135	0.60	3,402,000	3,402,000	5,103,000	6,804,000	6,804,000	6,804,000	6,804,000	6,804,000	6,804,000	6,804,000	6,804,000	4,094,806	2,131,329	0
Subtotal Operating	dollars	1,381,697,415		6,100,530	22,070,659	93,485,911	132,327,785	132,518,772	148,362,033	150,708,236	150,804,512	149,192,926	135,004,964	146,982,901	81,992,675	32,145,509	0
Unit Cost																	
Operating Cost per tonne milled	$/tonne ore	$ 11.35		$ -	$ -	$ 9.89	$ 10.50	$ 10.52	$ 11.77	$ 11.96	$ 11.97	$ 11.84	$ 10.71	$ 11.67	$ 10.81	$ 8.14	$ -
Mining Cost per Tonne Milled	$/tonne ore	$ 5.25		$ -	$ -	$ 3.85	$ 4.46	$ 4.48	$ 5.73	$ 5.92	$ 5.93	$ 5.80	$ 4.67	$ 5.63	$ 4.77	$ 2.10	$ -
Capital Cost																	
Open Pit Mining	dollars	118,539,000		36,867,000	18,500,000	23,196,000	0	20,815,000	5,077,000	7,396,000	1,000,000	315,000	4,800,000	573,000	0	0	0
Underground Mining	dollars	0		0	0	0	0	0	0	0	0	0	0	0	0	0	0
Processing	dollars	241,188,000		71,640,000	119,400,000	47,760,000	0	238,800	238,800	477,600	238,800	477,600	238,800	238,800	238,800	0	0
Infrastructure	dollars	140,900,000		23,800,000	68,655,000	4,355,000	13,090,000	3,100,000	3,100,000	3,100,000	3,100,000	7,440,000	3,100,000	3,100,000	3,100,000	1,860,000	0
Environment Costs	dollars	16,200,000		16,200,000	0	0	0	0	0	0	0	0	0	0	0	0	0
Indirect	dollars	118,603,052		37,222,696	58,086,526	20,863,830	0	0	0	0	0	0	0	729,000	972,000	729,000	0
Contingency	dollars	77,524,050		26,084,385	37,547,025	11,462,640	0	0	0	0	0	0	0	729,000	972,000	729,000	0
Subtotal Capital	dollars	712,954,102		211,814,081	302,188,551	107,637,470	13,090,000	24,153,800	8,415,800	10,973,600	4,338,800	8,232,600	8,138,800	5,369,800	5,282,800	3,318,000	0
Revenue (after refining)																	
Gold Gross Revenue	dollars	2,516,793,300		-	-	181,678,430	266,935,241	325,006,076	263,284,750	287,832,155	239,617,706	229,755,120	238,594,837	253,283,069	136,574,918	94,230,998	-
less Gold Refining	dollars	15,511,823		-	-	1,119,744	1,645,209	2,003,119	1,622,710	1,774,004	1,476,843	1,416,056	1,470,538	1,561,067	841,756	580,777	-
less Gold Payables	dollars	37,751,899		-	-	2,725,176	4,004,029	4,875,091	3,949,271	4,317,482	3,594,266	3,446,327	3,578,923	3,799,246	2,048,624	1,413,465	-
Subtotal		2,570,057,022		-	-	177,833,510	261,286,003	318,127,866	257,712,769	281,740,669	234,546,598	224,892,737	233,545,376	247,922,756	133,684,538	92,236,756	-
less Royalty	dollars	24,635,296		-	-	1,778,335	2,612,860	3,181,279	2,577,128	2,817,407	2,345,466	2,248,927	2,335,454	2,479,228	1,336,845	922,368	-
Net Revenue	dollars	2,438,894,281		-	-	176,055,175	258,673,143	314,946,588	255,135,641	278,923,262	232,201,132	222,643,809	231,209,922	245,443,528	132,347,693	91,314,389	-
Cashflow																	
Operating Cost	dollars	1,381,700,000		6,101,000	22,071,000	93,486,000	132,328,000	132,519,000	148,362,000	150,708,000	150,805,000	149,193,000	135,005,000	146,983,000	81,993,000	32,146,000	0
Capital Cost	dollars	712,956,000		211,814,000	302,189,000	107,637,000	13,090,000	24,154,000	8,416,000	10,974,000	4,339,000	8,233,000	8,139,000	5,370,000	5,283,000	3,318,000	0
Revenue	dollars	2,438,895,000		0	0	176,055,000	258,673,000	314,947,000	255,136,000	278,923,000	232,201,000	222,644,000	231,210,000	245,444,000	132,348,000	91,314,000	0
Net Cashflow	dollars	344,239,000		-217,915,000	-324,260,000	-25,068,000	113,255,000	158,274,000	98,358,000	117,241,000	77,057,000	65,218,000	88,066,000	93,091,000	45,072,000	55,850,000	0
Cumulative	dollars			-217,915,000	-542,175,000	-567,243,000	-453,988,000	-295,714,000	-197,356,000	-80,115,000	-3,058,000	62,160,000	150,226,000	243,317,000	288,389,000	344,239,000	0

NPV (millions) @	**0%**	$344
	5%	$108
	8%	$15
IRR		8.6%

40K Cashflow

Engineering Base

Plant Throughput	**40,000**	**tonnes per day**															
		Total		**Year -2**	**Year -1**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	**Year 11**	**Year 12**
Mill Production																	
Mill Feed	tonnes	126,851,384		-	-	10,800,000	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	9,124,611	6,126,773	-	-
Gold Grade	g/t	0.53		-	-	0.50	0.63	0.58	0.56	0.52	0.44	0.47	0.50	0.49	0.58	-	-
Gold Recovery	%			0.0%	0.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	0.0%	0.0%
Insitu Gold	ounces	2,152,410		-	-	174,473	293,169	266,950	261,160	241,438	205,805	218,511	233,401	142,443	115,060	-	-
Recovered Gold	ounces	1,937,169		-	-	157,026	263,852	240,255	235,044	217,295	185,225	196,660	210,061	128,198	103,554	-	-
Mine Production																	
Open Pit																	
Ore																	
Ore to Mill	tonnes	121,887,345		-	-	7,008,662	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	7,951,910	6,126,773	-	-
Ore to Stockpile	tonnes	4,964,039		115,405	4,848,634	-	-	-	-	-	-	-	-	-	-	-	-
Stockpile to Mill	tonnes	4,964,039		-	-	3,791,338	-	-	-	-	-	-	-	1,172,701	-	-	-
Waste	tonnes	215,762,364		960,274	8,435,298	13,823,059	16,799,286	21,243,140	29,106,757	30,501,996	28,338,778	25,269,990	24,301,459	15,199,651	1,782,676	-	-
Total Material	tonnes	347,577,787		1,075,679	13,283,932	24,623,059	31,199,286	35,643,140	43,506,757	44,901,996	42,738,778	39,669,990	38,701,459	24,324,262	7,909,449	-	-
Strip Ratio		1.70		-	-	1.28	1.17	1.48	2.02	2.12	1.97	1.75	1.69	1.67	0.29	-	-
Operating Cost																	
Open Pit Mining	dollars	650,022,003	5.12	2,033,033	25,106,631	41,267,622	58,966,651	67,365,535	82,227,771	84,864,772	80,776,290	74,976,282	73,145,758	44,342,801	14,948,859	0	0
Processing	dollars	688,803,015	5.43	0	0	58,644,000	78,192,000	78,192,000	78,192,000	78,192,000	78,192,000	78,192,000	78,192,000	49,546,638	33,268,377	0	0
G&A	dollars	73,450,720	0.58	3,744,000	3,744,000	5,616,000	7,488,000	7,488,000	7,488,000	7,488,000	7,488,000	7,488,000	7,488,000	4,744,798	3,185,922	0	0
Subtotal Operating	dollars	1,412,275,738		5,777,033	28,850,631	105,527,622	144,646,651	153,045,535	167,907,771	170,544,772	166,456,290	160,656,282	158,825,758	98,634,236	51,403,158	0	0
Unit Cost																	
Operating Cost per tonne milled	$/tonne ore	$ 11.13		$ -	$ -	$ 9.77	$ 10.04	$ 10.63	$ 11.66	$ 11.84	$ 11.56	$ 11.16	$ 11.03	$ 10.81	$ 8.39	$ -	$ -
Mining Cost per Tonne Milled	$/tonne ore	$ 5.12		$ -	$ -	$ 3.82	$ 4.09	$ 4.68	$ 5.71	$ 5.89	$ 5.61	$ 5.21	$ 5.08	$ 4.86	$ 2.44	$ -	$ -
Capital Cost																	
Open Pit Mining	dollars	127,366,000		38,167,000	21,500,000	24,696,000	12,600,000	13,115,000	6,777,000	6,896,000	2,500,000	1,115,000	0	0	0	0	0
Underground Mining	dollars	0		0	0	0	0	0	0	0	0	0	0	0	0	0	0
Processing	dollars	263,105,000		78,150,000	130,250,000	52,100,000	0	260,500	260,500	521,000	521,000	521,000	260,500	260,500	0	0	0
Infrastructure	dollars	140,900,000		23,800,000	68,655,000	4,355,000	13,090,000	3,100,000	3,100,000	3,100,000	3,100,000	7,440,000	3,100,000	3,100,000	4,960,000	0	0
Environment Costs	dollars	16,200,000		16,200,000	0	0	0	0	0	0	0	0	0	0	0	0	0
Indirect	dollars	126,148,520		39,662,876	61,859,260	22,196,384	0	0	0	0	0	0	729,000	972,000	729,000	0	0
Contingency	dollars	82,135,650		27,732,675	39,852,825	12,120,150	0	0	0	0	0	0	729,000	972,000	729,000	0	0
Subtotal Capital	dollars	755,855,170		223,712,551	322,117,085	115,467,534	25,690,000	16,475,500	10,137,500	10,517,000	6,121,000	9,076,000	4,818,500	5,304,500	6,418,000	0	0
Revenue (after refining)																	
Gold Gross Revenue	dollars	2,024,341,833		-	-	164,091,763	275,725,751	251,066,138	245,621,222	227,072,824	193,559,931	205,509,756	219,513,250	133,967,414	108,213,784	-	-
less Gold Refining	dollars	17,047,089		-	-	1,381,825	2,321,901	2,114,241	2,068,389	1,912,192	1,629,978	1,730,608	1,848,533	1,128,147	911,274	-	-
less Gold Payables	dollars	30,365,127		-	-	2,461,376	4,135,886	3,765,992	3,684,318	3,406,092	2,903,399	3,082,646	3,292,699	2,009,511	1,623,207	-	-
Subtotal		2,071,754,049		-	-	160,248,561	269,267,964	245,185,905	239,868,514	221,754,539	189,026,553	200,696,502	214,372,018	130,829,756	105,679,304	-	-
less Royalty	dollars	19,769,296		-	-	1,602,486	2,692,680	2,451,859	2,398,685	2,217,545	1,890,266	2,006,965	2,143,720	1,308,298	1,056,793	-	-
Net Revenue	dollars	1,957,160,320		-	-	158,646,076	266,575,284	242,734,046	237,469,829	219,536,994	187,136,288	198,689,536	212,228,298	129,521,458	104,622,511	-	-
Cashflow																	
Operating Cost	dollars	1,412,277,000		5,777,000	28,851,000	105,528,000	144,647,000	153,046,000	167,908,000	170,545,000	166,456,000	160,656,000	158,826,000	98,634,000	51,403,000	0	0
Capital Cost	dollars	755,858,000		223,713,000	322,117,000	115,468,000	25,690,000	16,476,000	10,138,000	10,517,000	6,121,000	9,076,000	4,819,000	5,305,000	6,418,000	0	0
Revenue	dollars	1,957,160,000		0	0	158,646,000	266,575,000	242,734,000	237,470,000	219,537,000	187,136,000	198,690,000	212,228,000	129,521,000	104,623,000	0	0
Net Cashflow	dollars	-210,975,000		-229,490,000	-350,968,000	-62,350,000	96,238,000	73,212,000	59,424,000	38,475,000	14,559,000	28,958,000	48,583,000	25,582,000	46,802,000	0	0
Cumulative	dollars			-229,490,000	-580,458,000	-642,808,000	-546,570,000	-473,358,000	-413,934,000	-375,459,000	-360,900,000	-331,942,000	-283,359,000	-257,777,000	-210,975,000	0	0

NPV (millions) @	**0%**	-$211
	5%	-$283
	8%	-$308
IRR		-6.8%

Financial Base

Plant Throughput	40,000	tonnes per day															
		Total		Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Mill Production																	
Mill Feed	tonnes	126,851,384		-	-	10,800,000	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	9,124,611	6,126,773	-	-
Gold Grade	g/t	0.53		-	-	0.50	0.63	0.58	0.56	0.52	0.44	0.47	0.50	0.49	0.58	-	-
Gold Recovery	%			0.0%	0.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	0.0%	0.0%
Insitu Gold	ounces	2,152,410		-	-	174,473	293,169	266,950	261,160	241,438	205,805	218,511	233,401	142,443	115,060	-	-
Recovered Gold	ounces	1,937,169		-	-	157,026	263,852	240,255	235,044	217,295	185,225	196,660	210,061	128,198	103,554	-	-
Mine Production																	
Open Pit																	
Ore																	
Ore to Mill	tonnes	121,887,345		-	-	7,008,662	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	7,951,910	6,126,773	-	-
Ore to Stockpile	tonnes	4,964,039		115,405	4,848,634	-	-	-	-	-	-	-	-	-	-	-	-
Stockpile to Mill	tonnes	4,964,039		-	-	3,791,338	-	-	-	-	-	-	-	1,172,701	-	-	-
Waste	tonnes	215,762,364		960,274	8,435,298	13,823,059	16,799,286	21,243,140	29,106,757	30,501,996	28,338,778	25,269,990	24,301,459	15,199,651	1,782,676	-	-
Total Material	tonnes	347,577,787		1,075,679	13,283,932	24,623,059	31,199,286	35,643,140	43,506,757	44,901,996	42,738,778	39,669,990	38,701,459	24,324,262	7,909,449	-	-
Strip Ratio		1.70		-	-	1.28	1.17	1.48	2.02	2.12	1.97	1.75	1.69	1.67	0.29	-	-
Operating Cost																	
Open Pit Mining	dollars	650,022,003	5.12	2,033,033	25,106,631	41,267,622	58,966,651	67,365,535	82,227,771	84,864,772	80,776,290	74,976,282	73,145,758	44,342,801	14,948,859	0	0
Processing	dollars	688,803,015	5.43	0	0	58,644,000	78,192,000	78,192,000	78,192,000	78,192,000	78,192,000	78,192,000	78,192,000	49,546,638	33,268,377	0	0
G&A	dollars	73,450,720	0.58	3,744,000	3,744,000	5,616,000	7,488,000	7,488,000	7,488,000	7,488,000	7,488,000	7,488,000	7,488,000	4,744,798	3,185,922	0	0
Subtotal Operating	dollars	1,412,275,738		5,777,033	28,850,631	105,527,622	144,646,651	153,045,535	167,907,771	170,544,772	166,456,290	160,656,282	158,825,758	98,634,236	51,403,158	0	0
Unit Cost																	
Operating Cost per tonne milled	$/tonne ore	$ 11.13		$ -	$ -	$ 9.77	$ 10.04	$ 10.63	$ 11.66	$ 11.84	$ 11.56	$ 11.16	$ 11.03	$ 10.81	$ 8.39	$ -	$ -
Mining Cost per Tonne Milled	$/tonne ore	$ 5.12		$ -	$ -	$ 3.82	$ 4.09	$ 4.68	$ 5.71	$ 5.89	$ 5.61	$ 5.21	$ 5.08	$ 4.86	$ 2.44	$ -	$ -
Capital Cost																	
Open Pit Mining	dollars	127,366,000		38,167,000	21,500,000	24,696,000	12,600,000	13,115,000	6,777,000	6,896,000	2,500,000	1,115,000	0	0	0	0	0
Underground Mining	dollars	0		0	0	0	0	0	0	0	0	0	0	0	0	0	0
Processing	dollars	263,105,000		78,150,000	130,250,000	52,100,000	0	260,500	260,500	521,000	521,000	521,000	260,500	260,500	0	0	0
Infrastructure	dollars	140,900,000		23,800,000	68,655,000	4,355,000	13,090,000	3,100,000	3,100,000	3,100,000	3,100,000	7,440,000	3,100,000	3,100,000	4,960,000	0	0
Environment Costs	dollars	16,200,000		16,200,000	0	0	0	0	0	0	0	0	0	0	0	0	0
Indirect	dollars	126,148,520		39,662,876	61,859,260	22,196,384	0	0	0	0	0	0	729,000	972,000	729,000	0	0
Contingency	dollars	82,135,650		27,732,675	39,852,825	12,120,150	0	0	0	0	0	0	729,000	972,000	729,000	0	0
Subtotal Capital	dollars	755,855,170		223,712,551	322,117,085	115,467,534	25,690,000	16,475,500	10,137,500	10,517,000	6,121,000	9,076,000	4,818,500	5,304,500	6,418,000	0	0
Revenue (after refining)																	
Gold Gross Revenue	dollars	2,589,879,014		-	-	209,933,820	352,754,819	321,206,089	314,240,034	290,509,800	247,634,463	262,922,692	280,838,320	171,393,678	138,445,298	-	-
less Gold Refining	dollars	15,962,274		-	-	1,293,891	2,174,144	1,979,699	1,936,764	1,790,507	1,526,252	1,620,479	1,730,899	1,056,355	853,284	-	-
less Gold Payables	dollars	38,848,185		-	-	3,149,007	5,291,322	4,818,091	4,713,601	4,357,647	3,714,517	3,943,840	4,212,575	2,570,905	2,076,679	-	-
Subtotal		2,644,689,474		-	-	205,490,922	345,289,353	314,408,299	307,589,669	284,361,646	242,393,694	257,358,373	274,894,846	167,766,418	135,515,335	-	-
less Royalty	dollars	25,350,686		-	-	2,054,909	3,452,894	3,144,083	3,075,897	2,843,616	2,423,937	2,573,584	2,748,948	1,677,664	1,355,153	-	-
Net Revenue	dollars	2,509,717,869		-	-	203,436,012	341,836,459	311,264,216	304,513,773	281,518,029	239,969,757	254,784,789	272,145,898	166,088,754	134,160,182	-	-
Cashflow																	
Operating Cost	dollars	1,412,277,000		5,777,000	28,851,000	105,528,000	144,647,000	153,046,000	167,908,000	170,545,000	166,456,000	160,656,000	158,826,000	98,634,000	51,403,000	0	0
Capital Cost	dollars	755,858,000		223,713,000	322,117,000	115,468,000	25,690,000	16,476,000	10,138,000	10,517,000	6,121,000	9,076,000	4,819,000	5,305,000	6,418,000	0	0
Revenue	dollars	2,509,718,000		0	0	203,436,000	341,836,000	311,264,000	304,514,000	281,518,000	239,970,000	254,785,000	272,146,000	166,089,000	134,160,000	0	0
Net Cashflow	dollars	341,583,000		-229,490,000	-350,968,000	-17,560,000	171,499,000	141,742,000	126,468,000	100,456,000	67,393,000	85,053,000	108,501,000	62,150,000	76,339,000	0	0
Cumulative	dollars			-229,490,000	-580,458,000	-598,018,000	-426,519,000	-284,777,000	-158,309,000	-57,853,000	9,540,000	94,593,000	203,094,000	265,244,000	341,583,000	0	0

NPV (millions) @	**0%**	$342
	5%	$112
	8%	$20
IRR		8.8%

20K Capital

	Unit Cost $	Unit Life	Total Capital Cost $Cdn	Units Required													
				Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Total Capital Cost																	
Open Pit Mining Capital			$ 83,884,000														
Underground Capital			$ -														
Processing Capital			$ 167,660,000														
Infrastructure Capital			$ 140,900,000														
Environmental Costs			$ 16,200,000														
Indirect Costs			$ 92,785,040														
Contingency			$ 61,296,600														
Total			$ 562,725,640														
Open Pit Mining Capital	**Unit Cost**	**Operating Life**	**Fleet Cost**														
Production Drill	$ 1,500,000	25,000 hrs	$ 7,500,000	1		1			1		1			1			
Loader - 11.5 m3	$ 1,600,000	35,000 hrs	$ -														
Loader - 17 m3	$ 4,900,000	35,000 hrs	$ 9,800,000	1						1							
Loader - 20 m3 (L-1350)	$ 3,300,000	35,000 hrs	$ -														
Hydraulic Shovel - 17 m3 O&K 120C	$ 4,800,000	60,000 hrs	$ -														
Hydraulic Shovel - 21 m3 O&K 170C	$ 6,200,000	60,000 hrs	$ 6,200,000		1												
Hydraulic Shovel - 28 m3 O&K 200C	$ 10,000,000	60,000 hrs	$ -														
Hydraulic Shovel - 34 m3 O&K 340C	$ 12,600,000	60,000 hrs	$ -														
Breaker Loader - 6.5 cubic metre	$ 1,000,000	20,000 hrs	$ 2,000,000	1							1						
Haulage Trucks (240 ton)	$ 3,750,000	60,000 hrs	$ -														
Haulage Trucks (200 ton)	$ 3,200,000	60,000 hrs	$ 41,600,000	3	4	3							1	2			
Haulage Trucks (150 ton)	$ 2,750,000	60,000 hrs	$ -														
Haulage Trucks (100 ton)	$ 1,800,000	60,000 hrs	$ -														
			$ -														
Tracked Dozer (433 kW)	$ 1,500,000	35,000 hrs	$ -														
Tracked Dozer (306 kW)	$ 1,100,000	35,000 hrs	$ 6,600,000	3									3				
Tracked Dozer (231 kW)	$ 875,000	35,000 hrs	$ -														
Grader (233 kW)	$ 800,000	20,000 hrs	$ 1,600,000	1								1					
Rubber Tired Dozer (350 kW)	$ 1,300,000	30,000 hrs	$ 2,600,000	1											1		
Utility Backhoe with hammer (2.3 cubic metre)	$ 505,000	10 years	$ 1,010,000	1											1		
Water Truck (Sterling)	$ 290,000	10 years	$ 580,000	1											1		
Tool Carrier	$ 350,000	10 years	$ 700,000	1											1		
Blasting Skid Steer Loader	$ 65,000	5 years	$ 195,000	1					1					1			
Light Plants	$ 17,000	4 years	$ 357,000	7				7				7					
Lube/Fuel Truck	$ 310,000	6 years	$ 310,000	1													
Mechanics Truck	$ 230,000	4 years	$ -														
Welding Truck	$ 220,000	6 years	$ -														
Crewcab Pickups	$ 52,000	2 years	$ 624,000	2		2		2		2		2		2			
Blasters Truck	$ 52,000	5 years	$ 156,000	1					1					1			
Pumps	$ 45,000	5 years	$ 540,000	4					4					4			
Pickup Truck	$ 46,000	2 years	$ 552,000	2		2		2		2		2		2			
Manbus	$ 80,000	5 years	$ 240,000	1					1					1			
Ambulance	$ 100,000	10 years	$ 100,000	1													
Fire Truck	$ 260,000	10 years	$ 260,000	1													
Compactor	$ 260,000	10 years	$ 260,000	1													
Lowboy	$ 100,000		$ 100,000	1													
	$ -	$ -	$ -														
Open Pit Mining Capital			**$ 83,884,000**														
Underground Capital																	
Equipment Capital			$ -														
Development Capital			$ -														
Ventilation, OP, WP Capital			$ -														
Underground Mining Capital			**$ -**														
Processing Capital																	
Plant	$ 166,000,000		$ 166,000,000	0.3	0.5	0.2											
			$ -														
			$ -														
Sustaining Capital (@1% of initial capital)	$ 1,660,000		$ 1,660,000					0.1	0.1	0.2	0.1	0.1	0.1	0.2	0.1		
Subtotal Processing Capital			$ 167,660,000														
Infrastructure Capital																	
Power - Power line upgrade	$ 3,750,000		$ 3,750,000	1													
Power - Electrical Substations	$ 10,000,000		$ 10,000,000	1													
Power - Pit Powerlines	$ 800,000		$ 800,000		1												
Explosives Storage Area	$ 250,000		$ 250,000	1													
Haul Road Construction	$ 3,500,000		$ 3,150,000	0.5	0.4												
Fuel Storage	$ 250,000		$ 250,000	1													
Shop and Garage	$ 6,000,000		$ 6,000,000	0.75	0.25												
Fresh Water and Pumping System	$ 2,000,000		$ 2,000,000	0.3	0.5	0.2											
			$ -														
Owners Cost	$ -		$ -	0.5	0.5												
Mobile Equipment	$ 1,000,000		$ 1,000,000	0.5	0.5												
Communications	$ 100,000		$ 100,000	1													
Office	$ 3,500,000		$ 3,500,000		0.75	0.25											
Access Road to Plant	$ 2,100,000		$ 2,100,000	1													
Construction Camp			$ -														
			$ -														
Tailings Management Area - Initial	$ 77,000,000		$ 77,000,000		0.79	0.04	0.17										
Tailings Management Area - Sustaining	$ 31,000,000		$ 31,000,000					0.07	0.07	0.07	0.07	0.24	0.15	0.10	0.10	0.10	0.05
			$ -														
Subtotal Infrastructure Capital	$ 141,250,000		$ 140,900,000														
Environmental Costs																	
Fisheries Compensation	$ 16,200,000		$ 16,200,000	1.000													
Interest on Financial Assurance			$ -														
			$ -														
Subtotal Environmental Costs			$ 16,200,000														
Indirect Costs																	
Open Pit Mining Indirects	$ 8,388,400		$ 8,388,400	0.5	0.5												
Underground Mining Indirects	$ -		$ -														
Processing Indirects	$ 50,968,640		$ 50,968,640	0.3	0.5	0.2											
Infrastructure Indirects	$ 30,998,000		$ 30,998,000	0.3	0.5	0.2											
Environmental Indirects	$ 2,430,000		$ 2,430,000												0.3	0.4	0.3
Subtotal Indirect Costs			$ 92,785,040														
Contingency																	
Open Pit Contingency	$ 12,582,600		$ 12,582,600	0.5	0.5												
Underground Contingency	$ -		$ -														
Processing Contingency	$ 25,149,000		$ 25,149,000	0.3	0.5	0.2											
Infrastructure Contingency	$ 21,135,000		$ 21,135,000	0.3	0.5	0.2											
Environmental Contingency	$ 2,430,000		$ 2,430,000												0.3	0.4	0.3
Subtotal Contingency			$ 61,296,600														

25K Capital

	Unit Cost $	Unit Life	Total Capital Cost $Cdn	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11
				Units Required												
Total Capital Cost																
Open Pit Mining Capital			$ 79,939,000													
Underground Capital			$ -													
Processing Capital			$ 193,819,000													
Infrastructure Capital			$ 140,900,000													
Environmental Costs			$ 16,200,000													
Indirect Costs			$ 100,342,876													
Contingency			$ 64,628,700													
Total			$ 595,829,576													
Open Pit Mining Capital	**Unit Cost**	**Operating Life**	**Fleet Cost**													
Production Drill	$ 1,500,000	25,000 hrs	$ 6,000,000	1	1				1	1						
Loader - 11.5 m3	$ 1,600,000	35,000 hrs	$ -													
Loader - 17 m3	$ 4,900,000	35,000 hrs	$ 9,800,000	1						1						
Loader - 20 m3 (L-1350)	$ 3,300,000	35,000 hrs	$ -													
Hydraulic Shovel - 17 m3 O&K 120C	$ 4,800,000	60,000 hrs	$ -													
Hydraulic Shovel - 21 m3 O&K 170C	$ 6,200,000	60,000 hrs	$ 6,200,000		1											
Hydraulic Shovel - 28 m3 O&K 200C	$ 10,000,000	60,000 hrs	$ -													
Hydraulic Shovel - 34 m3 O&K 340C	$ 12,600,000	60,000 hrs	$ -													
Breaker Loader - 6.5 cubic metre	$ 1,000,000	20,000 hrs	$ 2,000,000	1							1					
Haulage Trucks (240 ton)	$ 3,750,000	60,000 hrs	$ -													
Haulage Trucks (200 ton)	$ 3,200,000	60,000 hrs	$ 41,600,000	4	4	3			1	1						
Haulage Trucks (150 ton)	$ 2,750,000	60,000 hrs	$ -													
Haulage Trucks (100 ton)	$ 1,800,000	60,000 hrs	$ -													
			$ -													
Tracked Dozer (433 kW)	$ 1,500,000	35,000 hrs	$ -													
Tracked Dozer (306 kW)	$ 1,100,000	35,000 hrs	$ 6,600,000	3									3			
Tracked Dozer (231 kW)	$ 875,000	35,000 hrs	$ -													
Grader (233 kW)	$ 800,000	20,000 hrs	$ 1,600,000	1								1				
Rubber Tired Dozer (350 kW)	$ 1,300,000	30,000 hrs	$ 1,300,000	1												
Utility Backhoe with hammer (2.3 cubic metre)	$ 505,000	10 years	$ 505,000	1												
Water Truck (Sterling)	$ 290,000	10 years	$ 290,000	1												
Tool Carrier	$ 350,000	10 years	$ 350,000	1												
Blasting Skid Steer Loader	$ 65,000	5 years	$ 195,000	1					1					1		
Light Plants	$ 17,000	4 years	$ 357,000	7				7				7				
Lube/Fuel Truck	$ 310,000	6 years	$ 310,000	1												
Mechanics Truck	$ 230,000	4 years	$ -													
Welding Truck	$ 220,000	6 years	$ -													
Crewcab Pickups	$ 52,000	2 years	$ 624,000	2		2		2		2		2		2		
Blasters Truck	$ 52,000	5 years	$ 156,000	1					1					1		
Pumps	$ 45,000	5 years	$ 540,000	4					4					4		
Pickup Truck	$ 46,000	2 years	$ 552,000	2		2		2		2		2		2		
Manbus	$ 80,000	5 years	$ 240,000	1					1					1		
Ambulance	$ 100,000	10 years	$ 100,000	1												
Fire Truck	$ 260,000	10 years	$ 260,000	1												
Compactor	$ 260,000	10 years	$ 260,000	1												
Lowboy	$ 100,000		$ 100,000	1												
	$ -	$ -	$ -													
Open Pit Mining Capital			**$ 79,939,000**													
Underground Capital																
Equipment Capital			$ -													
Development Capital			$ -													
Ventilation, OP, WP Capital			$ -													
Underground Mining Capital			**$ -**													
Processing Capital																
Plant	$ 191,900,000		$ 191,900,000	0.3	0.5	0.2										
			$ -													
			$ -													
Sustaining Capital (@1% of initial capital)	$ 1,919,000		$ 1,919,000					0.1	0.1	0.2	0.1	0.2	0.1	0.1	0.1	
Subtotal Processing Capital			$ 193,819,000													
Infrastructure Capital																
Power - Power line upgrade	$ 3,750,000		$ 3,750,000	1												
Power - Electrical Substations	$ 10,000,000		$ 10,000,000	1												
Power - Pit Powerlines	$ 800,000		$ 800,000		1											
Explosives Storage Area	$ 250,000		$ 250,000	1												
Haul Road Construction	$ 3,500,000		$ 3,150,000	0.5	0.4											
Fuel Storage	$ 250,000		$ 250,000	1												
Shop and Garage	$ 6,000,000		$ 6,000,000	0.75	0.25											
Fresh Water and Pumping System	$ 2,000,000		$ 2,000,000	0.3	0.5	0.2										
			$ -													
Owners Cost	$ -		$ -	0.5	0.5											
Mobile Equipment	$ 1,000,000		$ 1,000,000	0.5	0.5											
Communications	$ 100,000		$ 100,000	1												
Office	$ 3,500,000		$ 3,500,000		0.75	0.25										
Access Road to Plant	$ 2,100,000		$ 2,100,000	1												
Construction Camp			$ -													
			$ -													
Tailings Management Area - Initial	$ 77,000,000		$ 77,000,000		0.79	0.04	0.17									
Tailings Management Area - Sustaining	$ 31,000,000		$ 31,000,000					0.09	0.09	0.08	0.08	0.24	0.15	0.09	0.09	0.09
			$ -													
Subtotal Infrastructure Capital	$ 141,250,000		$ 140,900,000													
Environmental Costs																
Fisheries Compensation	$ 16,200,000		$ 16,200,000	1.000												
Interest on Financial Assurance			$ -													
			$ -													
Subtotal Environmental Costs			$ 16,200,000													
Indirect Costs																
Open Pit Mining Indirects	$ 7,993,900		$ 7,993,900	0.5	0.5											
Underground Mining Indirects	$ -		$ -													
Processing Indirects	$ 58,920,976		$ 58,920,976	0.3	0.5	0.2										
Infrastructure Indirects	$ 30,998,000		$ 30,998,000	0.3	0.5	0.2										
Environmental Indirects	$ 2,430,000		$ 2,430,000											0.3	0.4	0.3
Subtotal Indirect Costs			$ 100,342,876													
Contingency																
Open Pit Contingency	$ 11,990,850		$ 11,990,850	0.5	0.5											
Underground Contingency	$ -		$ -													
Processing Contingency	$ 29,072,850		$ 29,072,850	0.3	0.5	0.2										
Infrastructure Contingency	$ 21,135,000		$ 21,135,000	0.3	0.5	0.2										
Environmental Contingency	$ 2,430,000		$ 2,430,000											0.3	0.4	0.3
Subtotal Contingency			$ 64,628,700													

30K Capital

	Unit Cost $	Unit Life	Total Capital Cost $Cdn	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
				Units Required											
Total Capital Cost															
Open Pit Mining Capital			$ 95,166,000												
Underground Capital			$ -												
Processing Capital			$ 218,261,000												
Infrastructure Capital			$ 140,900,000												
Environmental Costs			$ 16,200,000												
Indirect Costs			$ 109,295,944												
Contingency			$ 70,579,050												
Total			$ 650,401,994												
Open Pit Mining Capital	**Unit Cost**	**Operating Life**	**Fleet Cost**												
Production Drill	$ 1,500,000	25,000 hrs	$ 6,000,000	1	1				1	1					
Loader - 11.5 m3	$ 1,600,000	35,000 hrs	$ -												
Loader - 17 m3	$ 4,900,000	35,000 hrs	$ 9,800,000	1						1					
Loader - 20 m3 (L-1350)	$ 3,300,000	35,000 hrs	$ -												
Hydraulic Shovel - 17 m3 O&K 120C	$ 4,800,000	60,000 hrs	$ -												
Hydraulic Shovel - 21 m3 O&K 170C	$ 6,200,000	60,000 hrs	$ 12,400,000		1		1								
Hydraulic Shovel - 28 m3 O&K 200C	$ 10,000,000	60,000 hrs	$ -												
Hydraulic Shovel - 34 m3 O&K 340C	$ 12,600,000	60,000 hrs	$ -												
Breaker Loader - 6.5 cubic metre	$ 1,000,000	20,000 hrs	$ 2,000,000		1						1				
Haulage Trucks (240 ton)	$ 3,750,000	60,000 hrs	$ -												
Haulage Trucks (200 ton)	$ 3,200,000	60,000 hrs	$ 51,200,000	5	4	2	3			2					
Haulage Trucks (150 ton)	$ 2,750,000	60,000 hrs	$ -												
Haulage Trucks (100 ton)	$ 1,800,000	60,000 hrs	$ -												
			$ -												
Tracked Dozer (433 kW)	$ 1,500,000	35,000 hrs	$ -												
Tracked Dozer (306 kW)	$ 1,100,000	35,000 hrs	$ 6,600,000	3							3				
Tracked Dozer (231 kW)	$ 875,000	35,000 hrs	$ -												
Grader (233 kW)	$ 800,000	20,000 hrs	$ 1,600,000	1								1			
Rubber Tired Dozer (350 kW)	$ 1,300,000	30,000 hrs	$ 1,300,000	1											
Utility Backhoe with hammer (2.3 cubic metre)	$ 505,000	10 years	$ 505,000	1											
Water Truck (Sterling)	$ 290,000	10 years	$ 290,000	1											
Tool Carrier	$ 350,000	10 years	$ 350,000	1											
Blasting Skid Steer Loader	$ 65,000	5 years	$ 130,000	1					1						
Light Plants	$ 17,000	4 years	$ 357,000	7				7				7			
Lube/Fuel Truck	$ 310,000	6 years	$ 310,000	1											
Mechanics Truck	$ 230,000	4 years	$ -												
Welding Truck	$ 220,000	6 years	$ -												
Crewcab Pickups	$ 52,000	2 years	$ 520,000	2		2		2		2		2			
Blasters Truck	$ 52,000	5 years	$ 104,000	1					1						
Pumps	$ 45,000	5 years	$ 360,000	4					4						
Pickup Truck	$ 46,000	2 years	$ 460,000	2		2		2		2		2			
Manbus	$ 80,000	5 years	$ 160,000	1					1						
Ambulance	$ 100,000	10 years	$ 100,000	1											
Fire Truck	$ 260,000	10 years	$ 260,000	1											
Compactor	$ 260,000	10 years	$ 260,000	1											
Lowboy	$ 100,000		$ 100,000	1											
	$ -	$ -	$ -												
Open Pit Mining Capital			**$ 95,166,000**												
Underground Capital															
Equipment Capital			$ -												
Development Capital			$ -												
Ventilation, OP, WP Capital			$ -												
Underground Mining Capital			**$ -**												
Processing Capital															
Plant	$ 216,100,000		$ 216,100,000	0.3	0.5	0.2									
			$ -												
			$ -												
Sustaining Capital (@1% of initial capital)	$ 2,161,000		$ 2,161,000					0.2	0.1	0.2	0.1	0.1	0.2	0.1	
Subtotal Processing Capital			$ 218,261,000												
Infrastructure Capital															
Power - Power line upgrade	$ 3,750,000		$ 3,750,000	1											
Power - Electrical Substations	$ 10,000,000		$ 10,000,000	1											
Power - Pit Powerlines	$ 800,000		$ 800,000		1										
Explosives Storage Area	$ 250,000		$ 250,000	1											
Haul Road Construction	$ 3,500,000		$ 3,150,000	0.5	0.4										
Fuel Storage	$ 250,000		$ 250,000	1											
Shop and Garage	$ 6,000,000		$ 6,000,000	0.75	0.25										
Fresh Water and Pumping System	$ 2,000,000		$ 2,000,000	0.3	0.5	0.2									
			$ -												
Owners Cost	$ -		$ -	0.5	0.5										
Mobile Equipment	$ 1,000,000		$ 1,000,000	0.5	0.5										
Communications	$ 100,000		$ 100,000	1											
Office	$ 3,500,000		$ 3,500,000		0.75	0.25									
Access Road to Plant	$ 2,100,000		$ 2,100,000	1											
Construction Camp			$ -												
			$ -												
Tailings Management Area - Initial	$ 77,000,000		$ 77,000,000		0.79	0.04	0.17								
Tailings Management Area - Sustaining	$ 31,000,000		$ 31,000,000					0.09	0.10	0.10	0.15	0.24	0.09	0.10	0.13
			$ -												
Subtotal Infrastructure Capital	$ 141,250,000		$ 140,900,000												
Environmental Costs															
Fisheries Compensation	$ 16,200,000		$ 16,200,000	1.000											
Interest on Financial Assurance			$ -												
			$ -												
Subtotal Environmental Costs			$ 16,200,000												
Indirect Costs															
Open Pit Mining Indirects	$ 9,516,600		$ 9,516,600	0.5	0.5										
Underground Mining Indirects	$ -		$ -												
Processing Indirects	$ 66,351,344		$ 66,351,344	0.3	0.5	0.2									
Infrastructure Indirects	$ 30,998,000		$ 30,998,000	0.3	0.5	0.2									
Environmental Indirects	$ 2,430,000		$ 2,430,000										0.3	0.4	0.3
Subtotal Indirect Costs			$ 109,295,944												
Contingency															
Open Pit Contingency	$ 14,274,900		$ 14,274,900	0.5	0.5										
Underground Contingency	$ -		$ -												
Processing Contingency	$ 32,739,150		$ 32,739,150	0.3	0.5	0.2									
Infrastructure Contingency	$ 21,135,000		$ 21,135,000	0.3	0.5	0.2									
Environmental Contingency	$ 2,430,000		$ 2,430,000										0.3	0.4	0.3
Subtotal Contingency			$ 70,579,050												

35K Capital

	Unit Cost		Total Capital	Units Required												
	$	Unit Life	Cost $Cdn	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11
Total Capital Cost																
Open Pit Mining Capital			$ 118,539,000													
Underground Capital			$ -													
Processing Capital			$ 241,188,000													
Infrastructure Capital			$ 140,900,000													
Environmental Costs			$ 16,200,000													
Indirect Costs			$ 118,603,052													
Contingency			$ 77,524,050													
Total			$ 712,954,102													
Open Pit Mining Capital	**Unit Cost**	**Operating Life**	**Fleet Cost**													
Production Drill	$ 1,500,000	25,000 hrs	$ 9,000,000	1	1			1	1	1			1			
Loader - 11.5 m3	$ 1,600,000	35,000 hrs	$ -													
Loader - 17 m3	$ 4,900,000	35,000 hrs	$ 9,800,000	1						1						
Loader - 20 m3 (L-1350)	$ 3,300,000	35,000 hrs	$ -													
Hydraulic Shovel - 17 m3 O&K 120C	$ 4,800,000	60,000 hrs	$ -													
Hydraulic Shovel - 21 m3 O&K 170C	$ 6,200,000	60,000 hrs	$ 18,600,000	1		1		1								
Hydraulic Shovel - 28 m3 O&K 200C	$ 10,000,000	60,000 hrs	$ -													
Hydraulic Shovel - 34 m3 O&K 340C	$ 12,600,000	60,000 hrs	$ -													
Breaker Loader - 6.5 cubic metre	$ 1,000,000	20,000 hrs	$ 2,000,000		1						1					
Haulage Trucks (240 ton)	$ 3,750,000	60,000 hrs	$ -													
Haulage Trucks (200 ton)	$ 3,200,000	60,000 hrs	$ 64,000,000	5	5	5		4	1							
Haulage Trucks (150 ton)	$ 2,750,000	60,000 hrs	$ -													
Haulage Trucks (100 ton)	$ 1,800,000	60,000 hrs	$ -													
			$ -													
Tracked Dozer (433 kW)	$ 1,500,000	35,000 hrs	$ -													
Tracked Dozer (306 kW)	$ 1,100,000	35,000 hrs	$ 6,600,000	3									3			
Tracked Dozer (231 kW)	$ 875,000	35,000 hrs	$ -													
Grader (233 kW)	$ 800,000	20,000 hrs	$ 2,400,000	1		1				1						
Rubber Tired Dozer (350 kW)	$ 1,300,000	30,000 hrs	$ 1,300,000	1												
Utility Backhoe with hammer (2.3 cubic metre)	$ 505,000	10 years	$ 505,000	1												
Water Truck (Sterling)	$ 290,000	10 years	$ 290,000	1												
Tool Carrier	$ 350,000	10 years	$ 350,000	1												
Blasting Skid Steer Loader	$ 65,000	5 years	$ 195,000	1					1					1		
Light Plants	$ 17,000	4 years	$ 357,000	7				7				7				
Lube/Fuel Truck	$ 310,000	6 years	$ 310,000	1												
Mechanics Truck	$ 230,000	4 years	$ -													
Welding Truck	$ 220,000	6 years	$ -													
Crewcab Pickups	$ 52,000	2 years	$ 624,000	2		2		2		2		2		2		
Blasters Truck	$ 52,000	5 years	$ 156,000	1					1					1		
Pumps	$ 45,000	5 years	$ 540,000	4					4					4		
Pickup Truck	$ 46,000	2 years	$ 552,000	2		2		2		2		2		2		
Manbus	$ 80,000	5 years	$ 240,000	1					1					1		
Ambulance	$ 100,000	10 years	$ 100,000	1												
Fire Truck	$ 260,000	10 years	$ 260,000	1												
Compactor	$ 260,000	10 years	$ 260,000	1												
Lowboy	$ 100,000		$ 100,000	1												
	$ -	$ -	$ -													
Open Pit Mining Capital			**$ 118,539,000**													
Underground Capital																
Equipment Capital			$ -													
Development Capital			$ -													
Ventilation, OP, WP Capital			$ -													
Underground Mining Capital			**$ -**													
Processing Capital																
Plant	$ 238,800,000		$ 238,800,000	0.3	0.5	0.2										
			$ -													
			$ -													
Sustaining Capital (@1% of initial capital)	$ 2,388,000		$ 2,388,000					0.1	0.1	0.2	0.1	0.2	0.1	0.1	0.1	
Subtotal Processing Capital			$ 241,188,000													
Infrastructure Capital																
Power - Power line upgrade	$ 3,750,000		$ 3,750,000	1												
Power - Electrical Substations	$ 10,000,000		$ 10,000,000	1												
Power - Pit Powerlines	$ 800,000		$ 800,000		1											
Explosives Storage Area	$ 250,000		$ 250,000	1												
Haul Road Construction	$ 3,500,000		$ 3,150,000	0.5	0.4											
Fuel Storage	$ 250,000		$ 250,000	1												
Shop and Garage	$ 6,000,000		$ 6,000,000	0.75	0.25											
Fresh Water and Pumping System	$ 2,000,000		$ 2,000,000	0.3	0.5	0.2										
			$ -													
Owners Cost	$ -		$ -	0.5	0.5											
Mobile Equipment	$ 1,000,000		$ 1,000,000	0.5	0.5											
Communications	$ 100,000		$ 100,000	1												
Office	$ 3,500,000		$ 3,500,000		0.75	0.25										
Access Road to Plant	$ 2,100,000		$ 2,100,000	1												
Construction Camp			$ -													
			$ -													
Tailings Management Area - Initial	$ 77,000,000		$ 77,000,000		0.79	0.04	0.17									
Tailings Management Area - Sustaining	$ 31,000,000		$ 31,000,000					0.10	0.10	0.10	0.10	0.24	0.10	0.10	0.10	0.06
			$ -													
Subtotal Infrastructure Capital	$ 141,250,000		$ 140,900,000													
Environmental Costs																
Fisheries Compensation	$ 16,200,000		$ 16,200,000	1.000												
Interest on Financial Assurance			$ -													
			$ -													
Subtotal Environmental Costs			$ 16,200,000													
Indirect Costs																
Open Pit Mining Indirects	$ 11,853,900		$ 11,853,900	0.5	0.5											
Underground Mining Indirects	$ -		$ -													
Processing Indirects	$ 73,321,152		$ 73,321,152	0.3	0.5	0.2										
Infrastructure Indirects	$ 30,998,000		$ 30,998,000	0.3	0.5	0.2										
Environmental Indirects	$ 2,430,000		$ 2,430,000											0.3	0.4	0.3
Subtotal Indirect Costs			$ 118,603,052													
Contingency																
Open Pit Contingency	$ 17,780,850		$ 17,780,850	0.5	0.5											
Underground Contingency	$ -		$ -													
Processing Contingency	$ 36,178,200		$ 36,178,200	0.3	0.5	0.2										
Infrastructure Contingency	$ 21,135,000		$ 21,135,000	0.3	0.5	0.2										
Environmental Contingency	$ 2,430,000		$ 2,430,000											0.3	0.4	0.3
Subtotal Contingency			$ 77,524,050													

40K Capital

	Unit Cost $ US	Unit Life	Total Capital Cost $Cdn	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
				Units Required											
Total Capital Cost															
Open Pit Mining Capital			$ 127,366,000												
Underground Capital			$ -												
Processing Capital			$ 263,105,000												
Infrastructure Capital			$ 140,900,000												
Environmental Costs			$ 16,200,000												
Indirect Costs			$ 126,148,520												
Contingency			$ 82,135,650												
Total			$ 755,855,170												
Open Pit Mining Capital	**Unit Cost**	**Operating Life**	**Fleet Cost**												
Production Drill	$ 1,500,000	25,000 hrs	$ 9,000,000	1	1	1			1	1	1				
Loader - 11.5 m3	$ 1,600,000	35,000 hrs	$ -												
Loader - 17 m3	$ 4,900,000	35,000 hrs	$ 9,800,000	1					1						
Loader - 20 m3 (L-1350)	$ 3,300,000	35,000 hrs	$ -												
Hydraulic Shovel - 17 m3 O&K 120C	$ 4,800,000	60,000 hrs	$ -												
Hydraulic Shovel - 21 m3 O&K 170C	$ 6,200,000	60,000 hrs	$ 18,600,000		1	1	1								
Hydraulic Shovel - 28 m3 O&K 200C	$ 10,000,000	60,000 hrs	$ -												
Hydraulic Shovel - 34 m3 O&K 340C	$ 12,600,000	60,000 hrs	$ -												
Breaker Loader - 6.5 cubic metre	$ 1,000,000	20,000 hrs	$ 2,000,000		1						1				
Haulage Trucks (240 ton)	$ 3,750,000	60,000 hrs	$ -												
Haulage Trucks (200 ton)	$ 3,200,000	60,000 hrs	$ 70,400,000	7	4	5	2	4							
Haulage Trucks (150 ton)	$ 2,750,000	60,000 hrs	$ -												
Haulage Trucks (100 ton)	$ 1,800,000	60,000 hrs	$ -												
			$ -												
Tracked Dozer (433 kW)	$ 1,500,000	35,000 hrs	$ -												
Tracked Dozer (306 kW)	$ 1,100,000	35,000 hrs	$ 8,800,000	4						4					
Tracked Dozer (231 kW)	$ 875,000	35,000 hrs	$ -												
Grader (233 kW)	$ 800,000	20,000 hrs	$ 3,200,000	1		1				1		1			
Rubber Tired Dozer (350 kW)	$ 1,300,000	30,000 hrs	$ 1,300,000	1											
Utility Backhoe with hammer (2.3 cubic metre)	$ 505,000	10 years	$ 505,000	1											
Water Truck (Sterling)	$ 290,000	10 years	$ 290,000	1											
Tool Carrier	$ 350,000	10 years	$ 350,000	1											
Blasting Skid Steer Loader	$ 65,000	5 years	$ 130,000	1					1						
Light Plants	$ 17,000	4 years	$ 357,000	7				7				7			
Lube/Fuel Truck	$ 310,000	6 years	$ 310,000	1											
Mechanics Truck	$ 230,000	4 years	$ -												
Welding Truck	$ 220,000	6 years	$ -												
Crewcab Pickups	$ 52,000	2 years	$ 520,000	2		2		2		2		2			
Blasters Truck	$ 52,000	5 years	$ 104,000	1					1						
Pumps	$ 45,000	5 years	$ 360,000	4					4						
Pickup Truck	$ 46,000	2 years	$ 460,000	2		2		2		2		2			
Manbus	$ 80,000	5 years	$ 160,000	1					1						
Ambulance	$ 100,000	10 years	$ 100,000	1											
Fire Truck	$ 260,000	10 years	$ 260,000	1											
Compactor	$ 260,000	10 years	$ 260,000	1											
Lowboy	$ 100,000		$ 100,000	1											
	$ -	$ -	$ -												
Open Pit Mining Capital			**$ 127,366,000**												
Underground Capital															
Equipment Capital			$ -												
Development Capital			$ -												
Ventilation, OP, WP Capital			$ -												
Underground Mining Capital			**$ -**												
Processing Capital															
Plant	$ 260,500,000		$ 260,500,000	0.3	0.5	0.2									
			$ -												
			$ -												
Sustaining Capital (@1% of initial capital)	$ 2,605,000		$ 2,605,000					0.1	0.1	0.2	0.2	0.2	0.1	0.1	
Subtotal Processing Capital			$ 263,105,000												
Infrastructure Capital															
Power - Power line upgrade	$ 3,750,000		$ 3,750,000	1											
Power - Electrical Substations	$ 10,000,000		$ 10,000,000	1											
Power - Pit Powerlines	$ 800,000		$ 800,000		1										
Explosives Storage Area	$ 250,000		$ 250,000	1											
Haul Road Construction	$ 3,500,000		$ 3,150,000	0.5	0.4										
Fuel Storage	$ 250,000		$ 250,000	1											
Shop and Garage	$ 6,000,000		$ 6,000,000	0.75	0.25										
Fresh Water and Pumping System	$ 2,000,000		$ 2,000,000	0.3	0.5	0.2									
			$ -												
Owners Cost	$ -		$ -	0.5	0.5										
Mobile Equipment	$ 1,000,000		$ 1,000,000	0.5	0.5										
Communications	$ 100,000		$ 100,000	1											
Office	$ 3,500,000		$ 3,500,000		0.75	0.25									
Access Road to Plant	$ 2,100,000		$ 2,100,000	1											
Construction Camp			$ -												
			$ -												
Tailings Management Area - Initial	$ 77,000,000		$ 77,000,000		0.79	0.04	0.17								
Tailings Management Area - Sustaining	$ 31,000,000		$ 31,000,000					0.10	0.10	0.10	0.10	0.24	0.10	0.10	0.16
			$ -												
Subtotal Infrastructure Capital	$ 141,250,000		$ 140,900,000												
Environmental Costs															
Fisheries Compensation	$ 16,200,000		$ 16,200,000	1.000											
Interest on Financial Assurance			$ -												
			$ -												
Subtotal Environmental Costs			$ 16,200,000												
Indirect Costs															
Open Pit Mining Indirects	$ 12,736,600		$ 12,736,600	0.5	0.5										
Underground Mining Indirects	$ -		$ -												
Processing Indirects	$ 79,983,920		$ 79,983,920	0.3	0.5	0.2									
Infrastructure Indirects	$ 30,998,000		$ 30,998,000	0.3	0.5	0.2									
Environmental Indirects	$ 2,430,000		$ 2,430,000										0.3	0.4	0.3
Subtotal Indirect Costs			$ 126,148,520												
Contingency															
Open Pit Contingency	$ 19,104,900		$ 19,104,900	0.5	0.5										
Underground Contingency	$ -		$ -												
Processing Contingency	$ 39,465,750		$ 39,465,750	0.3	0.5	0.2									
Infrastructure Contingency	$ 21,135,000		$ 21,135,000	0.3	0.5	0.2									
Environmental Contingency	$ 2,430,000		$ 2,430,000										0.3	0.4	0.3
Subtotal Contingency			$ 82,135,650												

20K Mine Schedule

Mill		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Total
Mill Feed	Total	**TOTAL ORE TONNES**	**-**	**-**	**5,400,000**	**7,200,000**	**7,200,000**	**7,200,000**	**7,200,000**	**7,200,000**	**7,200,000**	**7,200,000**	**7,200,000**	**7,200,000**	**7,200,000**	**3,305,097**	**80,705,097**
		Au (g/t)	**-**	**-**	**0.48**	**0.50**	**0.66**	**0.79**	**0.54**	**0.66**	**0.72**	**0.60**	**0.53**	**0.59**	**0.53**	**0.48**	**0.60**
	From Mine	**High Grade**	-	-	4,722,907	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	3,305,097	80,028,004
		Au (g/t)	-	0.59	0.47	0.50	0.66	0.79	0.54	0.66	0.72	0.60	0.53	0.59	0.53	0.48	0.60
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	From Stockpile	**High Grade**	-	-	677,093	-	-	-	-	-	-	-	-	-	-	-	677,093
		Au (g/t)	-	-	0.59	-	-	-	-	-	-	-	-	-	-	-	0.59
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

Pit		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Total
Mine Production	Open Pit Ore	**High Grade**	-	677,093	4,722,907	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	3,305,097	80,705,097
		Au (g/t)	-	0.59	0.47	0.50	0.66	0.79	0.54	0.66	0.72	0.60	0.53	0.59	0.53	0.48	0.60
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL ORE TONNES	-	677,093	4,722,907	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	3,305,097	80,705,097
		Au (g/t)	-	0.59	0.47	0.50	0.66	0.79	0.54	0.66	0.72	0.60	0.53	0.59	0.53	0.48	0.60
	Open Pit Waste	**TOTAL WASTE TONNES**	-	4,038,543	9,206,311	10,908,793	11,292,373	12,111,315	12,301,289	11,300,840	12,642,345	19,168,064	9,749,014	5,811,564	5,429,385	603,609	124,563,443
		PAG (tonnes)	-	4,038,543	9,206,311	10,908,793	11,292,373	12,111,315	12,301,289	11,300,840	12,642,345	19,168,064	9,749,014	5,811,564	5,429,385	603,609	124,563,443
		NAG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Total Tonnes	-	4,715,636	13,929,218	18,108,793	18,492,373	19,311,315	19,501,289	18,500,840	19,842,345	26,368,064	16,949,014	13,011,564	12,629,385	3,908,706	205,268,540
		S.R.	0.0	6.0	1.9	1.5	1.6	1.7	1.7	1.6	1.8	2.7	1.4	0.8	0.8	0.2	1.5

		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Total
Direct to Mill	Open Pit to Mill	**High Grade**			4,722,907	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	3,305,097	80,028,004
		Au (g/t)	-	0.59	0.47	0.50	0.66	0.79	0.54	0.66	0.72	0.60	0.53	0.59	0.53	0.48	0.60
		Low Grade		-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL ORE TONNES	-	-	4,722,907	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	3,305,097	80,028,004
		Au (g/t)	-	-	0.47	0.50	0.66	0.79	0.54	0.66	0.72	0.60	0.53	0.59	0.53	0.48	0.60

		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Total
To Stockpile	Pit to Stockpile	**High Grade**	-	677,093	-	-	-	-	-	-	-	-	-	-	-	-	677,093
		Au (g/t)	-	0.59	0.47	0.50	0.66	0.79	0.54	0.66	0.72	0.60	0.53	0.59	0.53	0.48	0.59
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL ORE TONNES	-	677,093	-	-	-	-	-	-	-	-	-	-	-	-	677,093
		Au (g/t)	-	0.59	-	-	-	-	-	-	-	-	-	-	-	-	0.59
	Total	**TOTAL ORE TONNES**	-	677,093	-	-	-	-	-	-	-	-	-	-	-	-	677,093
		Au (g/t)	-	0.59	-	-	-	-	-	-	-	-	-	-	-	-	0.59

		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
	From Stockpile	**TOTAL ORE TONNES**	-	-	677,093	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	0.59	-	-	-	-	-	-	-	-	-	-	-
		High Grade			677,093											
		Au (g/t)	-	-	0.59	-	-	-	-	-	-	-	-	-	-	-
		Low Grade														
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Stockpile Inventory	**TOTAL ORE TONNES**	-	677,093	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	0.59	-	-	-	-	-	-	-	-	-	-	-	-
		High Grade	-	677,093	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	0.59	-	-	-	-	-	-	-	-	-	-	-	-
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-

Cut Summary

	Report File	Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Total
Phase 1		**High Grade**	-	70,984	1,070,511	3,120,325	4,691,076	3,640,410	236,258	580,035	-	-	-	-	-	-	13,409,599
		Au (g/t)	-	1.00	0.67	0.63	0.76	1.10	1.03	0.80	-	-	-	-	-	-	0.822
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL ORE TONNES	-	70,984	1,070,511	3,120,325	4,691,076	3,640,410	236,258	580,035	-	-	-	-	-	-	13,409,599
		OB (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		AG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		PAG (tonnes)	-	354,132	2,797,555	4,104,727	3,915,930	1,814,289	254,287	379,208	-	-	-	-	-	-	13,620,128
		NAG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL WASTE TONNES	-	354,132	2,797,555	4,104,727	3,915,930	1,814,289	254,287	379,208	-	-	-	-	-	-	13,620,128
		Total Tonnes	-	425,116	3,868,066	7,225,052	8,607,006	5,454,699	490,545	959,243	-	-	-	-	-	-	27,029,727
		S.R.	-	5.0	2.6	1.3	0.8	0.5	1.1	0.7	-	-	-	-	-	-	1.0
Phase 2		**High Grade**	-	453,894	1,827,543	1,914,216	2,086,832	1,931,273	4,857,137	4,665,711	5,072,804	4,333,861	4,390,331	4,212,919	3,246,168	-	38,992,689
		Au (g/t)	-	0.58	0.47	0.44	0.49	0.50	0.58	0.72	0.80	0.68	0.53	0.58	0.62	-	0.612
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Total Ore Tonnes	-	453,894	1,827,543	1,914,216	2,086,832	1,931,273	4,857,137	4,665,711	5,072,804	4,333,861	4,390,331	4,212,919	3,246,168	-	38,992,689
		OB (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		AG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		PAG (tonnes)	-	1,259,503	3,263,666	3,090,637	5,098,830	4,983,007	6,823,153	8,367,976	7,499,407	9,874,574	5,190,754	2,455,090	250,200	-	58,156,797
		NAG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL WASTE TONNES	-	1,259,503	3,263,666	3,090,637	5,098,830	4,983,007	6,823,153	8,367,976	7,499,407	9,874,574	5,190,754	2,455,090	250,200	-	58,156,797
		Total Tonnes	-	1,713,397	5,091,209	5,004,853	7,185,662	6,914,280	11,680,290	13,033,687	12,572,211	14,208,435	9,581,085	6,668,009	3,496,368	-	97,149,486
		S.R.	-	2.8	1.8	1.6	2.4	2.6	1.4	1.8	1.5	2.3	1.2	0.6	0.1	-	1.5
Phase 3		**High Grade**	-	152,215	1,824,853	2,165,459	422,092	1,628,317	2,106,605	1,954,254	2,127,196	2,866,139	2,809,669	2,987,081	3,953,832	3,305,097	28,302,809
		Au (g/t)	-	0.46	0.35	0.37	0.45	0.44	0.38	0.48	0.52	0.48	0.54	0.62	0.45	0.48	0.472
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Total Ore Tonnes	-	152,215	1,824,853	2,165,459	422,092	1,628,317	2,106,605	1,954,254	2,127,196	2,866,139	2,809,669	2,987,081	3,953,832	3,305,097	28,302,809
		OB (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		AG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		PAG (tonnes)	-	2,424,908	3,145,090	3,713,429	2,277,613	5,314,019	5,223,849	2,553,656	5,142,938	9,293,490	4,558,260	3,356,474	5,179,185	603,609	52,786,518
		NAG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL WASTE TONNES	-	2,424,908	3,145,090	3,713,429	2,277,613	5,314,019	5,223,849	2,553,656	5,142,938	9,293,490	4,558,260	3,356,474	5,179,185	603,609	52,786,518
		Total Tonnes	-	2,577,123	4,969,943	5,878,888	2,699,705	6,942,336	7,330,454	4,507,910	7,270,134	12,159,629	7,367,929	6,343,555	9,133,017	3,908,706	81,089,327
		S.R.	-	15.9	1.7	1.7	5.4	3.3	2.5	1.3	2.4	3.2	1.6	1.1	1.3	0.2	1.9

25K Mine Schedule

Mill		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Total
Mill Feed	Total	**TOTAL ORE TONNES**	**-**	**-**	**6,750,000**	**9,000,000**	**9,000,000**	**9,000,000**	**9,000,000**	**9,000,000**	**9,000,000**	**9,000,000**	**9,000,000**	**6,093,697**	**4,093,899**	**-**	**88,937,596**
		Au (g/t)	**-**	**-**	**0.51**	**0.51**	**0.73**	**0.64**	**0.64**	**0.65**	**0.53**	**0.51**	**0.53**	**0.55**	**0.43**	**-**	**0.57**
	From Mine	**High Grade**	-	-	5,632,599	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	6,093,697	4,093,899	-	87,820,195
		Au (g/t)	-	0.57	0.50	0.51	0.73	0.64	0.64	0.65	0.53	0.51	0.53	0.55	0.43	-	0.57
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	From Stockpile	**High Grade**	-	-	1,117,401	-	-	-	-	-	-	-	-	-	-	-	1,117,401
		Au (g/t)	-	-	0.57	-	-	-	-	-	-	-	-	-	-	-	0.57
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

Pit		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Total
Mine Production	**Open Pit Ore**	**High Grade**	-	1,117,401	5,632,599	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	6,093,697	4,093,899	-	88,937,596
		Au (g/t)	-	0.57	0.50	0.51	0.73	0.64	0.64	0.65	0.53	0.51	0.53	0.55	0.43	-	0.57
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL ORE TONNES	-	1,117,401	5,632,599	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	6,093,697	4,093,899	-	88,937,596
		Au (g/t)	-	0.57	0.50	0.51	0.73	0.64	0.64	0.65	0.53	0.51	0.53	0.55	0.43	-	0.57
	Open Pit Waste	**TOTAL WASTE TONNES**	-	4,766,910	9,711,382	11,882,769	11,137,134	11,643,620	14,599,589	16,033,360	16,159,364	15,479,628	9,901,105	6,384,894	1,729,508	-	129,429,262
		PAG (tonnes)	-	4,766,910	9,711,382	11,882,769	11,137,134	11,643,620	14,599,589	16,033,360	16,159,364	15,479,628	9,901,105	6,384,894	1,729,508	-	129,429,262
		NAG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Total Tonnes	-	5,884,311	15,343,981	20,882,769	20,137,134	20,643,620	23,599,589	25,033,360	25,159,364	24,479,628	18,901,105	12,478,591	5,823,407	-	218,366,858
		S.R.	0.0	4.3	1.7	1.3	1.2	1.3	1.6	1.8	1.8	1.7	1.1	1.0	0.4	0.0	1.5

		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Total
Direct to Mill	Open Pit to Mill	**High Grade**			5,632,599	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	6,093,697	4,093,899	-	87,820,195
		Au (g/t)	-	0.57	0.50	0.51	0.73	0.64	0.64	0.65	0.53	0.51	0.53	0.55	0.43	-	0.57
		Low Grade		-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL ORE TONNES	-	-	5,632,599	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	9,000,000	6,093,697	4,093,899	-	87,820,195
		Au (g/t)	-	-	0.50	0.51	0.73	0.64	0.64	0.65	0.53	0.51	0.53	0.55	0.43	-	0.57

		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Total
To Stockpile	Pit to Stockpile	**High Grade**	-	1,117,401	-	-	-	-	-	-	-	-	-	-	-	-	1,117,401
		Au (g/t)	-	0.57	0.50	0.51	0.73	0.64	0.64	0.65	0.53	0.51	0.53	0.55	0.43	-	0.57
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL ORE TONNES	-	1,117,401	-	-	-	-	-	-	-	-	-	-	-	-	1,117,401
		Au (g/t)	-	0.57	-	-	-	-	-	-	-	-	-	-	-	-	0.57
	Total	**TOTAL ORE TONNES**	-	1,117,401	-	-	-	-	-	-	-	-	-	-	-	-	1,117,401
		Au (g/t)	-	0.57	-	-	-	-	-	-	-	-	-	-	-	-	0.57

	Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
From Stockpile	**TOTAL ORE TONNES**	-	-	1,117,401	-	-	-	-	-	-	-	-	-	-	-
	Au (g/t)	-	-	0.57	-	-	-	-	-	-	-	-	-	-	-
	High Grade			1,117,401											
	Au (g/t)	-	-	0.57	-	-	-	-	-	-	-	-	-	-	-
	Low Grade														
	Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Stockpile Inventory	**TOTAL ORE TONNES**	-	1,117,401	-	-	-	-	-	-	-	-	-	-	-	-
	Au (g/t)	-	0.57	-	-	-	-	-	-	-	-	-	-	-	-
	High Grade	-	1,117,401	-	-	-	-	-	-	-	-	-	-	-	-
	Au (g/t)	-	0.57	-	-	-	-	-	-	-	-	-	-	-	-
	Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-

Cut Summary

Report File	Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Total
Phase 1	**High Grade**	-	201,381	2,614,178	4,276,610	5,314,281	2,758,455	1,241,511	-	-	-	-	-	-	-	16,406,416
	Au (g/t)	-	0.82	0.60	0.65	0.92	0.98	0.76	-	-	-	-	-	-	-	0.795
	Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	TOTAL ORE TONNES	-	201,381	2,614,178	4,276,610	5,314,281	2,758,455	1,241,511	-	-	-	-	-	-	-	16,406,416
	OB (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	AG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	PAG (tonnes)	-	1,173,256	4,250,610	4,127,461	3,047,977	1,285,718	863,094	-	-	-	-	-	-	-	14,748,116
	NAG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	TOTAL WASTE TONNES	-	1,173,256	4,250,610	4,127,461	3,047,977	1,285,718	863,094	-	-	-	-	-	-	-	14,748,116
	Total Tonnes	-	1,374,637	6,864,788	8,404,071	8,362,258	4,044,173	2,104,605	-	-	-	-	-	-	-	31,154,532
	S.R.	-	5.8	1.6	1.0	0.6	0.5	0.7	-	-	-	-	-	-	-	0.9
Phase 2	**High Grade**	-	748,724	1,766,549	1,864,427	1,772,416	4,947,875	6,743,315	6,233,815	4,957,432	6,920,214	5,073,262	1,310,422	-	-	42,338,451
	Au (g/t)	-	0.53	0.46	0.43	0.46	0.51	0.66	0.75	0.58	0.53	0.55	0.59	-	-	0.581
	Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Total Ore Tonnes	-	748,724	1,766,549	1,864,427	1,772,416	4,947,875	6,743,315	6,233,815	4,957,432	6,920,214	5,073,262	1,310,422	-	-	42,338,451
	OB (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	AG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	PAG (tonnes)	-	1,501,905	2,689,711	4,137,921	3,143,015	6,219,963	9,553,967	10,599,709	9,159,963	7,489,406	1,135,283	62,550	-	-	55,693,393
	NAG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	TOTAL WASTE TONNES	-	1,501,905	2,689,711	4,137,921	3,143,015	6,219,963	9,553,967	10,599,709	9,159,963	7,489,406	1,135,283	62,550	-	-	55,693,393
	Total Tonnes	-	2,250,629	4,456,260	6,002,348	4,915,431	11,167,838	16,297,282	16,833,524	14,117,395	14,409,620	6,208,545	1,372,972	-	-	98,031,844
	S.R.	-	2.0	1.5	2.2	1.8	1.3	1.4	1.7	1.8	1.1	0.2	0.0	-	-	1.3
Phase 3	**High Grade**	-	167,296	1,251,872	2,858,963	1,913,303	1,293,670	1,015,174	2,766,185	4,042,568	2,079,786	3,926,738	4,783,275	4,093,899	-	30,192,729
	Au (g/t)	-	0.47	0.33	0.35	0.43	0.39	0.35	0.43	0.48	0.42	0.50	0.54	0.43	-	0.446
	Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Total Ore Tonnes	-	167,296	1,251,872	2,858,963	1,913,303	1,293,670	1,015,174	2,766,185	4,042,568	2,079,786	3,926,738	4,783,275	4,093,899	-	30,192,729
	OB (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	AG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	PAG (tonnes)	-	2,091,749	2,771,061	3,617,387	4,946,142	4,137,939	4,182,528	5,433,651	6,999,401	7,990,223	8,765,822	6,322,344	1,729,508	-	58,987,753
	NAG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	TOTAL WASTE TONNES	-	2,091,749	2,771,061	3,617,387	4,946,142	4,137,939	4,182,528	5,433,651	6,999,401	7,990,223	8,765,822	6,322,344	1,729,508	-	58,987,753
	Total Tonnes	-	2,259,045	4,022,933	6,476,350	6,859,445	5,431,609	5,197,702	8,199,836	11,041,969	10,070,009	12,692,560	11,105,619	5,823,407	-	89,180,482
	S.R.	-	12.5	2.2	1.3	2.6	3.2	4.1	2.0	1.7	3.8	2.2	1.3	0.4	-	2.0

30K Schedule

Mill		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
Mill Feed	**Total**	**TOTAL ORE TONNES**	**-**	**-**	**8,100,000**	**10,800,000**	**10,800,000**	**10,800,000**	**10,800,000**	**10,800,000**	**10,800,000**	**10,800,000**	**7,964,886**	**5,426,212**	**-**	**97,091,098**
		Au (g/t)	**-**	**-**	**0.46**	**0.48**	**0.71**	**0.65**	**0.65**	**0.56**	**0.47**	**0.51**	**0.57**	**0.43**	**-**	**0.56**
	From Mine	**High Grade**	-	-	6,158,418	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	7,964,886	5,426,212	-	95,149,516
		Au (g/t)	-	0.49	0.44	0.48	0.71	0.65	0.65	0.56	0.47	0.51	0.57	0.43	-	0.56
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	From Stockpile	**High Grade**	-	-	1,941,582	-	-	-	-	-	-	-	-	-	-	1,941,582
		Au (g/t)	-	-	0.49	-	-	-	-	-	-	-	-	-	-	0.49
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-

Pit		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
Mine Production	**Open Pit Ore**	**High Grade**	-	1,941,582	6,158,418	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	7,964,886	5,426,212	-	97,091,098
		Au (g/t)	-	0.49	0.44	0.48	0.71	0.65	0.65	0.56	0.47	0.51	0.57	0.43	-	0.56
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL ORE TONNES	-	1,941,582	6,158,418	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	7,964,886	5,426,212	-	97,091,098
		Au (g/t)	-	0.49	0.44	0.48	0.71	0.65	0.65	0.56	0.47	0.51	0.57	0.43	-	0.56
	Open Pit Waste	**TOTAL WASTE TONNES**	-	6,187,670	10,655,344	11,118,593	16,912,609	16,577,516	16,287,524	19,744,515	19,733,798	13,284,211	7,837,425	2,874,175	-	141,213,379
		PAG (tonnes)	-	6,187,670	10,655,344	11,118,593	16,912,609	16,577,516	16,287,524	19,744,515	19,733,798	13,284,211	7,837,425	2,874,175	-	141,213,379
		NAG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Total Tonnes	-	8,129,252	16,813,762	21,918,593	27,712,609	27,377,516	27,087,524	30,544,515	30,533,798	24,084,211	15,802,311	8,300,387	-	238,304,477
		S.R.	0.0	3.2	1.7	1.0	1.6	1.5	1.5	1.8	1.8	1.2	1.0	0.5	0.0	1.5

		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
Direct to Mill	Open Pit to Mill	**High Grade**			6,158,418	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	7,964,886	5,426,212	-	95,149,516
		Au (g/t)	-	0.49	0.44	0.48	0.71	0.65	0.65	0.56	0.47	0.51	0.57	0.43	-	0.56
		Low Grade			-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL ORE TONNES	-	-	6,158,418	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	10,800,000	7,964,886	5,426,212	-	95,149,516
		Au (g/t)	-	-	0.44	0.48	0.71	0.65	0.65	0.56	0.47	0.51	0.57	0.43	-	0.56

		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
To Stockpile	Pit to Stockpile	**High Grade**	-	1,941,582	-	-	-	-	-	-	-	-	-	-	-	1,941,582
		Au (g/t)	-	0.49	0.44	0.48	0.71	0.65	0.65	0.56	0.47	0.51	0.57	0.43	-	0.49
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL ORE TONNES	-	1,941,582	-	-	-	-	-	-	-	-	-	-	-	1,941,582
		Au (g/t)	-	0.49	-	-	-	-	-	-	-	-	-	-	-	0.49
	Total	**TOTAL ORE TONNES**	-	1,941,582	-	-	-	-	-	-	-	-	-	-	-	1,941,582
		Au (g/t)	-	0.49	-	-	-	-	-	-	-	-	-	-	-	0.49

	Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11
From Stockpile	**TOTAL ORE TONNES**	-	-	1,941,582	-	-	-	-	-	-	-	-	-	-
	Au (g/t)	-	-	0.49	-	-	-	-	-	-	-	-	-	-
	High Grade			1,941,582										
	Au (g/t)	-	-	0.49	-	-	-	-	-	-	-	-	-	-
	Low Grade													
	Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-
Stockpile Inventory	**TOTAL ORE TONNES**	-	1,941,582	-	-	-	-	-	-	-	-	-	-	-
	Au (g/t)	-	0.49	-	-	-	-	-	-	-	-	-	-	-
	High Grade	-	1,941,582	-	-	-	-	-	-	-	-	-	-	-
	Au (g/t)	-	0.49	-	-	-	-	-	-	-	-	-	-	-
	Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-
	Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-

Cut Summary

Report File		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
Phase 1		**High Grade**	-	210,203	2,009,766	4,114,874	6,588,102	2,966,389	2,373,504	-	-	-	-	-	-	18,262,838
		Au (g/t)	-	0.82	0.59	0.61	0.87	0.96	0.70	-	-	-	-	-	-	0.773
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL ORE TONNES	-	210,203	2,009,766	4,114,874	6,588,102	2,966,389	2,373,504	-	-	-	-	-	-	18,262,838
		OB (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		AG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		PAG (tonnes)	-	1,212,903	3,485,506	2,788,447	5,248,278	1,698,379	1,131,411	-	-	-	-	-	-	15,564,924
		NAG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL WASTE TONNES	-	1,212,903	3,485,506	2,788,447	5,248,278	1,698,379	1,131,411	-	-	-	-	-	-	15,564,924
		Total Tonnes	-	1,423,106	5,495,272	6,903,321	11,836,380	4,664,768	3,504,915	-	-	-	-	-	-	33,827,762
		S.R.	-	5.8	1.7	0.7	0.8	0.6	0.5	-	-	-	-	-	-	0.9
Phase 2		**High Grade**	-	1,292,866	1,727,841	3,503,620	2,854,563	6,467,618	6,301,911	5,736,784	7,597,128	6,486,819	3,477,780	-	-	45,446,930
		Au (g/t)	-	0.48	0.44	0.44	0.47	0.57	0.72	0.66	0.49	0.53	0.55	-	-	0.558
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Total Ore Tonnes	-	1,292,866	1,727,841	3,503,620	2,854,563	6,467,618	6,301,911	5,736,784	7,597,128	6,486,819	3,477,780	-	-	45,446,930
		OB (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		AG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		PAG (tonnes)	-	1,820,503	2,558,273	4,497,712	5,120,310	11,283,227	10,056,082	9,537,767	10,660,068	2,561,424	390,939	-	-	58,486,304
		NAG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL WASTE TONNES	-	1,820,503	2,558,273	4,497,712	5,120,310	11,283,227	10,056,082	9,537,767	10,660,068	2,561,424	390,939	-	-	58,486,304
		Total Tonnes	-	3,113,369	4,286,114	8,001,332	7,974,873	17,750,845	16,357,993	15,274,551	18,257,196	9,048,243	3,868,719	-	-	103,933,234
		S.R.	-	1.4	1.5	1.3	1.8	1.7	1.6	1.7	1.4	0.4	0.1	-	-	1.3
Phase 3		**High Grade**	-	438,513	2,420,811	3,181,506	1,357,335	1,365,993	2,124,585	5,063,216	3,202,872	4,313,181	4,487,106	5,426,212	-	33,381,330
		Au (g/t)	-	0.39	0.32	0.38	0.40	0.36	0.39	0.45	0.43	0.48	0.57	0.43	-	0.440
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Total Ore Tonnes	-	438,513	2,420,811	3,181,506	1,357,335	1,365,993	2,124,585	5,063,216	3,202,872	4,313,181	4,487,106	5,426,212	-	33,381,330
		OB (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		AG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		PAG (tonnes)	-	3,154,264	4,611,565	3,832,434	6,544,021	3,595,910	5,100,031	10,206,748	9,073,730	10,722,787	7,446,486	2,874,175	-	67,162,151
		NAG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL WASTE TONNES	-	3,154,264	4,611,565	3,832,434	6,544,021	3,595,910	5,100,031	10,206,748	9,073,730	10,722,787	7,446,486	2,874,175	-	67,162,151
		Total Tonnes	-	3,592,777	7,032,376	7,013,940	7,901,356	4,961,903	7,224,616	15,269,964	12,276,602	15,035,968	11,933,592	8,300,387	-	100,543,481
		S.R.	-	7.2	1.9	1.2	4.8	2.6	2.4	2.0	2.8	2.5	1.7	0.5	-	2.0

35K Schedule

Mill		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Total
Mill Feed	Total	**TOTAL ORE TONNES**	**-**	**-**	**9,450,000**	**12,600,000**	**12,600,000**	**12,600,000**	**12,600,000**	**12,600,000**	**12,600,000**	**12,600,000**	**12,600,000**	**7,582,974**	**3,946,905**	**-**	**121,779,879**
		Au (g/t)	**-**	**-**	**0.50**	**0.55**	**0.67**	**0.54**	**0.59**	**0.49**	**0.47**	**0.49**	**0.52**	**0.47**	**0.62**	**-**	**0.53**
	From Mine	**High Grade**	-	-	6,671,937	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	7,582,974	3,946,905	-	119,001,816
		Au (g/t)	0.94	0.53	0.48	0.55	0.67	0.54	0.59	0.49	0.47	0.49	0.52	0.47	0.62	-	0.53
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	From Stockpile	**High Grade**	-	-	2,778,063	-	-	-	-	-	-	-	-	-	-	-	2,778,063
		Au (g/t)	-	-	0.55	-	-	-	-	-	-	-	-	-	-	-	0.55
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

Pit		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Total
Mine Production	**Open Pit Ore**	**High Grade**	131,164	2,646,899	6,671,937	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	7,582,974	3,946,905	-	121,779,879
		Au (g/t)	0.94	0.53	0.48	0.55	0.67	0.54	0.59	0.49	0.47	0.49	0.52	0.47	0.62	-	0.53
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL ORE TONNES	131,164	2,646,899	6,671,937	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	7,582,974	3,946,905	-	121,779,879
		Au (g/t)	0.94	0.53	0.48	0.55	0.67	0.54	0.59	0.49	0.47	0.49	0.52	0.47	0.62	-	0.53
	Open Pit Waste	**TOTAL WASTE TONNES**	1,259,831	6,976,121	11,379,032	16,381,333	16,479,780	24,646,409	25,855,792	25,905,419	25,074,704	17,761,322	23,935,516	11,072,445	334,643	-	207,062,345
		PAG (tonnes)	1,259,831	6,976,121	11,379,032	16,381,333	16,479,780	24,646,409	25,855,792	25,905,419	25,074,704	17,761,322	23,935,516	11,072,445	334,643	-	207,062,345
		NAG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Total Tonnes	1,390,995	9,623,020	18,050,969	28,981,333	29,079,780	37,246,409	38,455,792	38,505,419	37,674,704	30,361,322	36,535,516	18,655,419	4,281,548	-	328,842,224
		S.R.	9.6	2.6	1.7	1.3	1.3	2.0	2.1	2.1	2.0	1.4	1.9	1.5	0.1	0.0	1.7

		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Total
Direct to Mill	Open Pit to Mill	**High Grade**			6,671,937	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	7,582,974	3,946,905	-	119,001,816
		Au (g/t)	0.94	0.53	0.48	0.55	0.67	0.54	0.59	0.49	0.47	0.49	0.52	0.47	0.62	-	0.53
		Low Grade			-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL ORE TONNES	-	-	6,671,937	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	12,600,000	7,582,974	3,946,905	-	119,001,816
		Au (g/t)	-	-	0.48	0.55	0.67	0.54	0.59	0.49	0.47	0.49	0.52	0.47	0.62	-	0.53

		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Total
To Stockpile	Pit to Stockpile	**High Grade**	131,164	2,646,899	-	-	-	-	-	-	-	-	-	-	-	-	2,778,063
		Au (g/t)	0.94	0.53	0.48	0.55	0.67	0.54	0.59	0.49	0.47	0.49	0.52	0.47	0.62	-	0.55
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL ORE TONNES	131,164	2,646,899	-	-	-	-	-	-	-	-	-	-	-	-	2,778,063
		Au (g/t)	0.94	0.53	-	-	-	-	-	-	-	-	-	-	-	-	0.55
	Total	**TOTAL ORE TONNES**	131,164	2,646,899	-	-	-	-	-	-	-	-	-	-	-	-	2,778,063
		Au (g/t)	0.94	0.53	-	-	-	-	-	-	-	-	-	-	-	-	0.55

		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
	From Stockpile	**TOTAL ORE TONNES**	-	-	2,778,063	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	0.55	-	-	-	-	-	-	-	-	-	-	-
		High Grade			2,778,063											
		Au (g/t)	-	-	0.55	-	-	-	-	-	-	-	-	-	-	-
		Low Grade														
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Stockpile Inventory	**TOTAL ORE TONNES**	131,164	2,778,063	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	0.94	0.55	-	-	-	-	-	-	-	-	-	-	-	-
		High Grade	131,164	2,778,063	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	0.94	0.55	-	-	-	-	-	-	-	-	-	-	-	-
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-

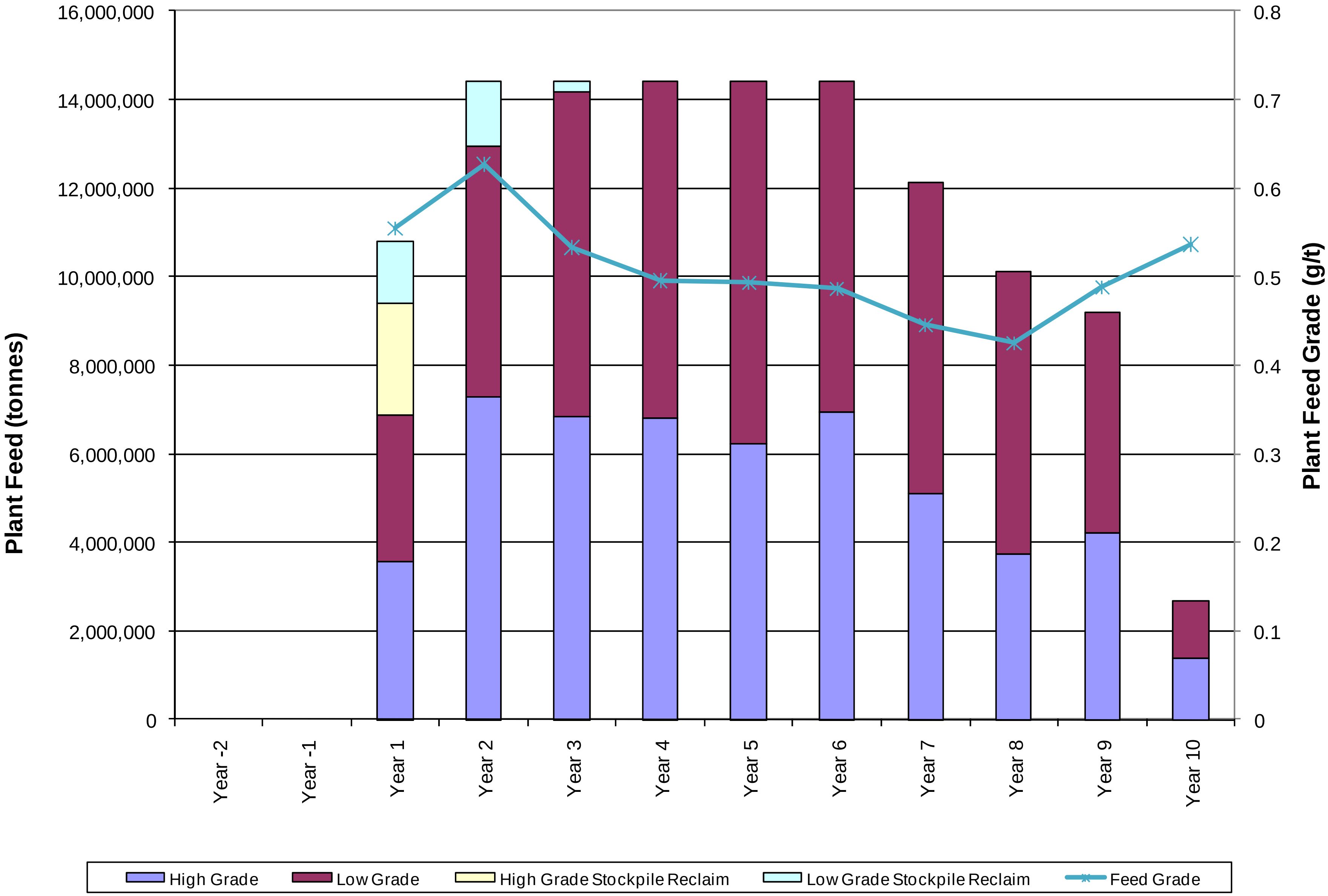
Cut Summary

Report File		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Total
Phase 1		**High Grade**	131,164	1,217,907	3,292,638	7,104,693	6,454,251	1,517,707	2,359,439	-	-	-	-	-	-	-	22,077,799
		Au (g/t)	0.94	0.60	0.57	0.69	0.89	1.02	0.70	-	-	-	-	-	-	-	0.752
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL ORE TONNES	131,164	1,217,907	3,292,638	7,104,693	6,454,251	1,517,707	2,359,439	-	-	-	-	-	-	-	22,077,799
		OB (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		AG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		PAG (tonnes)	869,299	2,555,702	4,887,799	5,517,666	3,761,611	1,481,949	800,238	-	-	-	-	-	-	-	19,874,264
		NAG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL WASTE TONNES	869,299	2,555,702	4,887,799	5,517,666	3,761,611	1,481,949	800,238	-	-	-	-	-	-	-	19,874,264
		Total Tonnes	1,000,463	3,773,609	8,180,437	12,622,359	10,215,862	2,999,656	3,159,677	-	-	-	-	-	-	-	41,952,063
		S.R.	6.6	2.1	1.5	0.8	0.6	1.0	0.3	-	-	-	-	-	-	-	0.9
Phase 2		**High Grade**	-	1,397,717	1,819,684	2,010,025	2,631,962	7,071,373	5,558,131	6,373,227	7,952,564	7,126,205	2,689,651	-	-	-	44,630,539
		Au (g/t)	-	0.46	0.44	0.42	0.44	0.55	0.70	0.57	0.49	0.52	0.54	-	-	-	0.537
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Total Ore Tonnes	-	1,397,717	1,819,684	2,010,025	2,631,962	7,071,373	5,558,131	6,373,227	7,952,564	7,126,205	2,689,651	-	-	-	44,630,539
		OB (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		AG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		PAG (tonnes)	390,532	1,529,940	2,555,960	2,558,049	5,236,790	9,680,011	10,692,883	10,977,152	10,353,561	2,467,599	215,798	-	-	-	56,658,274
		NAG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL WASTE TONNES	390,532	1,529,940	2,555,960	2,558,049	5,236,790	9,680,011	10,692,883	10,977,152	10,353,561	2,467,599	215,798	-	-	-	56,658,274
		Total Tonnes	390,532	2,927,657	4,375,644	4,568,074	7,868,752	16,751,384	16,251,014	17,350,379	18,306,125	9,593,804	2,905,449	-	-	-	101,288,813
		S.R.	-	1.1	1.4	1.3	2.0	1.4	1.9	1.7	1.3	0.3	0.1	-	-	-	1.3
Phase 3		**High Grade**	-	31,275	1,559,615	3,485,282	3,513,787	4,010,920	4,682,430	6,226,773	4,647,436	5,473,795	9,910,349	7,582,974	3,946,905	-	55,071,541
		Au (g/t)	-	0.50	0.32	0.34	0.42	0.35	0.40	0.41	0.44	0.45	0.51	0.47	0.62	-	0.445
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Total Ore Tonnes	-	31,275	1,559,615	3,485,282	3,513,787	4,010,920	4,682,430	6,226,773	4,647,436	5,473,795	9,910,349	7,582,974	3,946,905	-	55,071,541
		OB (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		AG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		PAG (tonnes)	-	2,890,479	3,935,273	8,305,618	7,481,378	13,484,449	14,362,671	14,928,267	14,721,143	15,293,723	23,719,718	11,072,445	334,643	-	130,529,807
		NAG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL WASTE TONNES	-	2,890,479	3,935,273	8,305,618	7,481,378	13,484,449	14,362,671	14,928,267	14,721,143	15,293,723	23,719,718	11,072,445	334,643	-	130,529,807
		Total Tonnes	-	2,921,754	5,494,888	11,790,900	10,995,165	17,495,369	19,045,101	21,155,040	19,368,579	20,767,518	33,630,067	18,655,419	4,281,548	-	185,601,348
		S.R.	-	92.4	2.5	2.4	2.1	3.4	3.1	2.4	3.2	2.8	2.4	1.5	0.1	-	2.4

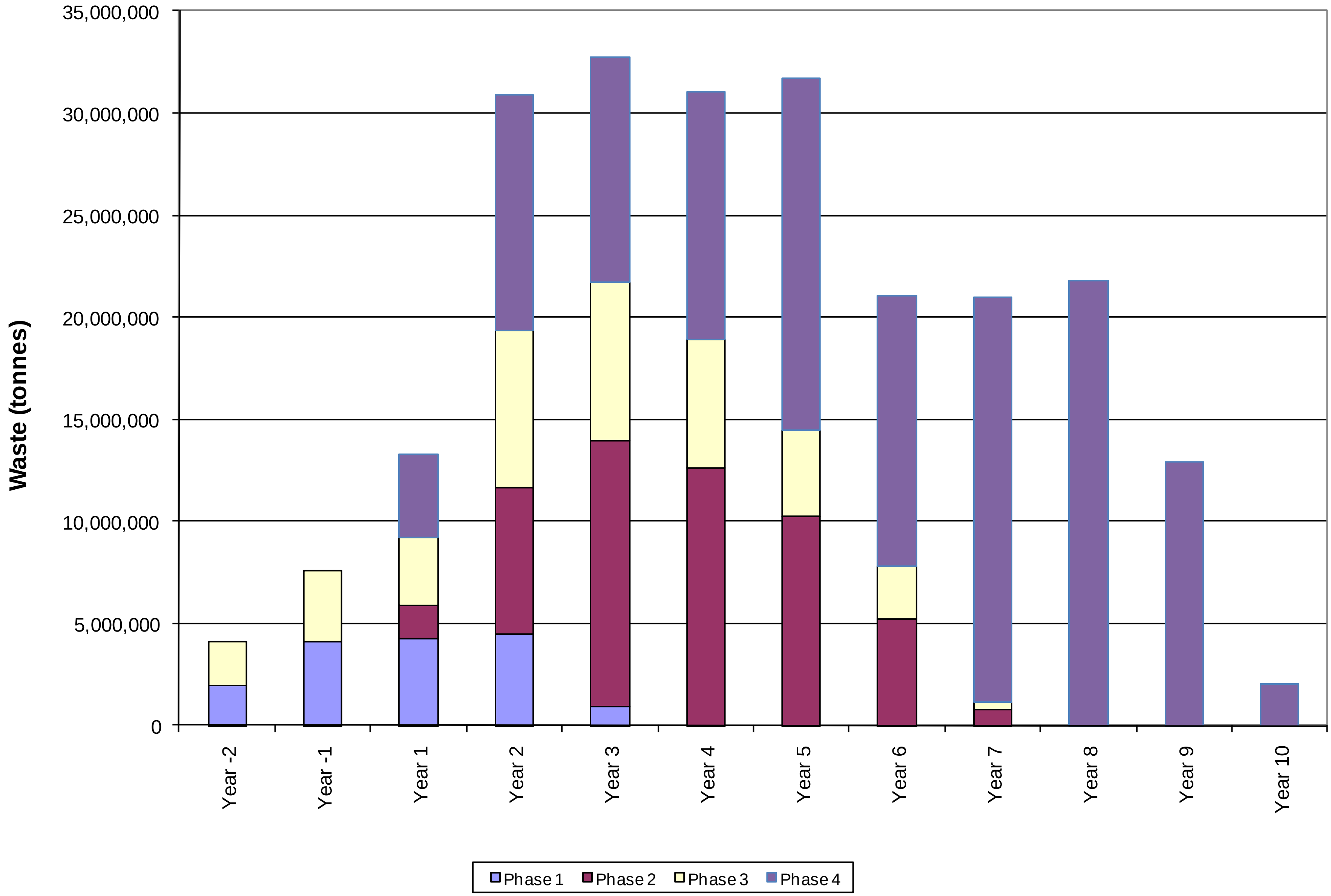
40K Schedule

Mill		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
Mill Feed	**Total**	**TOTAL ORE TONNES**	**-**	**-**	**10,800,000**	**14,400,000**	**14,400,000**	**14,400,000**	**14,400,000**	**14,400,000**	**14,400,000**	**14,400,000**	**9,124,611**	**6,126,773**	**-**	**126,851,384**
		Au (g/t)	**-**	**-**	**0.50**	**0.63**	**0.58**	**0.56**	**0.52**	**0.44**	**0.47**	**0.50**	**0.49**	**0.58**	**-**	**0.53**
	From Mine	**High Grade**	-	-	7,008,662	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	7,951,910	6,126,773	-	121,887,345
		Au (g/t)	0.90	0.50	0.50	0.63	0.58	0.56	0.52	0.44	0.47	0.50	0.48	0.58	-	0.53
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	From Stockpile	**High Grade**	-	-	3,791,338	-	-	-	-	-	-	-	1,172,701	-	-	4,964,039
		Au (g/t)	-	-	0.51	-	-	-	-	-	-	-	0.51	-	-	0.51
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Pit		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
Mine Production	**Open Pit Ore**	**High Grade**	115,405	4,848,634	7,008,662	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	7,951,910	6,126,773	-	126,851,384
		Au (g/t)	0.90	0.50	0.50	0.63	0.58	0.56	0.52	0.44	0.47	0.50	0.48	0.58	-	0.53
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL ORE TONNES	115,405	4,848,634	7,008,662	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	7,951,910	6,126,773	-	126,851,384
		Au (g/t)	0.90	0.50	0.50	0.63	0.58	0.56	0.52	0.44	0.47	0.50	0.48	0.58	-	0.53
	Open Pit Waste	**TOTAL WASTE TONNES**	960,274	8,435,298	13,823,059	16,799,286	21,243,140	29,106,757	30,501,996	28,338,778	25,269,990	24,301,459	15,199,651	1,782,676	-	215,762,364
		PAG (tonnes)	960,274	8,435,298	13,823,059	16,799,286	21,243,140	29,106,757	30,501,996	28,338,778	25,269,990	24,301,459	15,199,651	1,782,676	-	215,762,364
		NAG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Total Tonnes	1,075,679	13,283,932	20,831,721	31,199,286	35,643,140	43,506,757	44,901,996	42,738,778	39,669,990	38,701,459	23,151,561	7,909,449	-	342,613,748
		S.R.	8.3	1.7	2.0	1.2	1.5	2.0	2.1	2.0	1.8	1.7	1.9	0.3	0.0	1.7
		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
Direct to Mill	Open Pit to Mill	**High Grade**			7,008,662	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	7,951,910	6,126,773	-	121,887,345
		Au (g/t)	0.90	0.50	0.50	0.63	0.58	0.56	0.52	0.44	0.47	0.50	0.48	0.58	-	0.53
		Low Grade			-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL ORE TONNES	-	-	7,008,662	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	7,951,910	6,126,773	-	121,887,345
		Au (g/t)	-	-	0.50	0.63	0.58	0.56	0.52	0.44	0.47	0.50	0.48	0.58	-	0.53
		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
To Stockpile	Pit to Stockpile	**High Grade**	115,405	4,848,634	-	-	-	-	-	-	-	-	-	-	-	4,964,039
		Au (g/t)	0.90	0.50	0.50	0.63	0.58	0.56	0.52	0.44	0.47	0.50	0.48	0.58	-	0.51
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL ORE TONNES	115,405	4,848,634	-	-	-	-	-	-	-	-	-	-	-	4,964,039
		Au (g/t)	0.90	0.50	-	-	-	-	-	-	-	-	-	-	-	0.51
	Total	**TOTAL ORE TONNES**	115,405	4,848,634	-	-	-	-	-	-	-	-	-	-	-	4,964,039
		Au (g/t)	0.90	0.50	-	-	-	-	-	-	-	-	-	-	-	0.51
		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	
	From Stockpile	**TOTAL ORE TONNES**	-	-	3,791,338	-	-	-	-	-	-	-	1,172,701	-	-	
		Au (g/t)			0.51	-	-	-	-	-	-	-	0.51	-	-	
		High Grade			3,791,338								1,172,701			
		Au (g/t)	-	-	0.51	-	-	-	-	-	-	-	0.51	-	-	
		Low Grade														
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	
	Stockpile Inventory	**TOTAL ORE TONNES**	115,405	4,964,039	1,172,701	1,172,701	1,172,701	1,172,701	1,172,701	1,172,701	1,172,701	1,172,701	-	-	-	
		Au (g/t)	0.90	0.51	0.51	0.51	0.51	0.51	0.51	0.51	0.51	0.51	-	-	-	
		High Grade	115,405	4,964,039	1,172,701	1,172,701	1,172,701	1,172,701	1,172,701	1,172,701	1,172,701	1,172,701	-	-	-	
		Au (g/t)	0.90	0.51	0.51	0.51	0.51	0.51	0.51	0.51	0.51	0.51	-	-	-	
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	
Cut Summary																
	Report File	Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
Phase 1		**High Grade**	115,405	1,873,131	4,072,905	9,082,754	4,684,995	1,620,919	1,365,753	-	-	-	-	-	-	22,815,862
		Au (g/t)	0.90	0.59	0.58	0.78	0.88	0.76	0.74	-	-	-	-	-	-	0.745
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL ORE TONNES	115,405	1,873,131	4,072,905	9,082,754	4,684,995	1,620,919	1,365,753	-	-	-	-	-	-	22,815,862
		OB (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		AG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		PAG (tonnes)	418,766	4,008,931	5,930,698	5,261,588	2,671,876	1,844,463	147,957	-	-	-	-	-	-	20,284,279
		NAG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL WASTE TONNES	418,766	4,008,931	5,930,698	5,261,588	2,671,876	1,844,463	147,957	-	-	-	-	-	-	20,284,279
		Total Tonnes	534,171	5,882,062	10,003,603	14,344,342	7,356,871	3,465,382	1,513,710	-	-	-	-	-	-	43,100,141
		S.R.	3.6	2.1	1.5	0.6	0.6	1.1	0.1	-	-	-	-	-	-	0.9
Phase 2		**High Grade**	-	2,956,738	2,214,085	3,386,454	3,968,951	7,756,091	6,941,278	7,774,732	7,337,114	4,494,218	390,938	-	-	47,220,599
		Au (g/t)	-	0.45	0.41	0.43	0.50	0.66	0.58	0.48	0.51	0.52	0.47	-	-	0.525
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Total Ore Tonnes	-	2,956,738	2,214,085	3,386,454	3,968,951	7,756,091	6,941,278	7,774,732	7,337,114	4,494,218	390,938	-	-	47,220,599
		OB (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		AG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		PAG (tonnes)	541,508	2,784,767	4,495,768	6,295,580	6,420,120	10,918,553	12,588,539	11,194,615	3,099,354	641,139	6,255	-	-	58,986,198
		NAG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL WASTE TONNES	541,508	2,784,767	4,495,768	6,295,580	6,420,120	10,918,553	12,588,539	11,194,615	3,099,354	641,139	6,255	-	-	58,986,198
		Total Tonnes	541,508	5,741,505	6,709,853	9,682,034	10,389,071	18,674,644	19,529,817	18,969,347	10,436,468	5,135,357	397,193	-	-	106,206,797
		S.R.	-	0.9	2.0	1.9	1.6	1.4	1.8	1.4	0.4	0.1	0.0	-	-	1.2
Phase 3		**High Grade**	-	18,765	721,672	1,930,792	5,746,054	5,022,990	6,092,969	6,625,268	7,062,886	9,905,782	7,560,972	6,126,773	-	56,814,923
		Au (g/t)	-	0.51	0.30	0.31	0.39	0.35	0.40	0.40	0.44	0.50	0.48	0.58	-	0.443
		Low Grade	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Au (g/t)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		Total Ore Tonnes	-	18,765	721,672	1,930,792	5,746,054	5,022,990	6,092,969	6,625,268	7,062,886	9,905,782	7,560,972	6,126,773	-	56,814,923
		OB (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		AG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		PAG (tonnes)	-	1,641,600	3,396,593	5,242,118	12,151,144	16,343,741	17,765,500	17,144,163	22,170,636	23,660,320	15,193,396	1,782,676	-	136,491,887
		NAG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		TOTAL WASTE TONNES	-	1,641,600	3,396,593	5,242,118	12,151,144	16,343,741	17,765,500	17,144,163	22,170,636	23,660,320	15,193,396	1,782,676	-	136,491,887
		Total Tonnes	-	1,660,365	4,118,265	7,172,910	17,897,198	21,366,731	23,858,469	23,769,431	29,233,522	33,566,102	22,754,368	7,909,449	-	193,306,810
		S.R.	-	87.5	4.7	2.7	2.1	3.3	2.9	2.6	3.1	2.4	2.0	0.3	-	2.4

Final 40K Mining Schedule For PEA Report



Plant Feed (tonnes)
Plant Feed Grade (g/t)
16,000,000
14,000,000
12,000,000
10,000,000
8,000,000
6,000,000
4,000,000
2,000,000
0
0.8
0.7
0.6
0.5
0.4
0.3
0.2
0.1
0
Year -2
Year -1
Year 1
Year 2
Year 3
Year 4
Year 5
Year 6
Year 7
Year 8
Year 9
Year 10
High Grade
Low Grade
High Grade Stockpile Reclaim
Low Grade Stockpile Reclaim
Feed Grade



Waste (tonnes)
35,000,000
30,000,000
25,000,000
20,000,000
15,000,000
10,000,000
5,000,000
0
Year -2
Year -1
Year 1
Year 2
Year 3
Year 4
Year 5
Year 6
Year 7
Year 8
Year 9
Year 10
Phase 1
Phase 2
Phase 3
Phase 4

Mill		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
Mill Feed	Total	**TOTAL ORE TONNES**	**-**	**-**	**10,800,000**	**14,400,000**	**14,400,000**	**14,400,000**	**14,400,000**	**14,400,000**	**12,124,406**	**10,105,983**	**9,193,317**	**2,682,268**	**-**	**116,905,974**
		DAu (g/t)	**-**	**-**	**0.55**	**0.63**	**0.53**	**0.50**	**0.49**	**0.49**	**0.45**	**0.42**	**0.49**	**0.54**	**-**	**0.51**
	From Mine	**High Grade (Au>0.4g/t)**	-	-	3,549,683	7,279,532	6,835,634	6,822,014	6,223,122	6,940,133	5,093,536	3,726,495	4,223,924	1,396,148	-	52,090,220
		DAu (g/t)	0.65	0.72	0.82	0.97	0.82	0.74	0.78	0.71	0.68	0.69	0.74	0.77	-	0.78
		Low Grade (0.196<Au<0.4 g/t)	-	-	3,325,398	5,682,681	7,348,544	7,577,986	8,176,878	7,459,867	7,030,870	6,379,489	4,969,394	1,286,120	-	59,237,226
		DAu (g/t)	0.29	0.28	0.28	0.28	0.27	0.28	0.28	0.27	0.28	0.27	0.28	0.29	-	0.28
	From Stockpile	**High Grade (Au>0.4g/t)**	-	-	2,535,527	-	-	-	-	-	-	-	-	-	-	2,535,527
		DAu (g/t)	-	-	0.70	-	-	-	-	-	-	-	-	-	-	0.70
		Low Grade (0.196<Au<0.4 g/t)	-	-	1,389,392	1,437,787	215,822	-	-	-	-	-	-	-	-	3,043,001
		DAu (g/t)	-	-	0.28	0.28	0.28	-	-	-	-	-	-	-	-	0.28

Pit		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
Mine Production	Open Pit Ore	**High Grade (Au>0.4g/t)**	772,719	1,762,808	3,549,683	7,279,532	6,835,634	6,822,014	6,223,122	6,940,133	5,093,536	3,726,495	4,223,924	1,396,148	-	54,625,747
		DAu (g/t)	0.65	0.72	0.82	0.97	0.82	0.74	0.78	0.71	0.68	0.69	0.74	0.77	-	0.77
		Low Grade (0.196<Au<0.4 g/t)	873,923	2,169,078	3,325,398	5,682,681	7,348,544	7,577,986	8,176,878	7,459,867	7,030,870	6,379,489	4,969,394	1,286,120	-	62,280,227
		DAu (g/t)	0.29	0.28	0.28	0.28	0.27	0.28	0.28	0.27	0.28	0.27	0.28	0.29	-	0.28
		TOTAL ORE TONNES	1,646,642	3,931,886	6,875,081	12,962,213	14,184,178	14,400,000	14,400,000	14,400,000	12,124,406	10,105,983	9,193,317	2,682,268	-	116,905,974
		DAu (g/t)	0.46	0.48	0.56	0.67	0.54	0.50	0.49	0.49	0.45	0.42	0.49	0.54	-	0.51
	Open Pit Waste	**TOTAL WASTE TONNES**	4,088,038	7,590,346	13,272,973	30,896,338	32,744,747	31,067,578	31,682,172	21,054,746	21,002,742	21,785,538	12,915,230	2,025,713	-	230,126,161
		AG (tonnes)	-	-	-	-	3,128	-	12,510	-	-	-	-	-	-	15,638
		PAG (tonnes)	1,173,551	3,490,302	10,483,793	26,404,159	27,027,683	22,790,271	22,806,922	15,186,994	12,462,979	12,928,584	9,075,599	1,489,597	-	165,320,432
		NAG (tonnes)	2,914,487	4,100,044	2,789,180	4,492,179	5,713,936	8,277,308	8,862,741	5,867,753	8,539,763	8,856,955	3,839,632	536,116	-	64,790,091
		Total Tonnes	5,734,680	11,522,232	20,148,054	43,858,551	46,928,925	45,467,578	46,082,172	35,454,746	33,127,148	31,891,521	22,108,547	4,707,981	-	347,032,135
		S.R.	2.5	1.9	1.9	2.4	2.3	2.2	2.2	1.5	1.7	2.2	1.4	0.8	0.0	1.97

		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
Direct to Mill	Open Pit to Mill	**High Grade (Au>0.4g/t)**			3,549,683	7,279,532	6,835,634	6,822,014	6,223,122	6,940,133	5,093,536	3,726,495	4,223,924	1,396,148	-	52,090,220
		DAu (g/t)	0.65	0.72	0.82	0.97	0.82	0.74	0.78	0.71	0.68	0.69	0.74	0.77	-	0.78
		Au (g/t)	0.68	0.75	0.84	0.99	0.85	0.76	0.81	0.74	0.70	0.71	0.76	0.80	-	0.80
		Low Grade (0.196<Au<0.4 g/t)			3,325,398	5,682,681	7,348,544	7,577,986	8,176,878	7,459,867	7,030,870	6,379,489	4,969,394	1,286,120	-	59,237,226
		DAu (g/t)	0.29	0.28	0.28	0.28	0.27	0.28	0.28	0.27	0.28	0.27	0.28	0.29	-	0.28
		TOTAL ORE TONNES	-	-	6,875,081	12,962,213	14,184,178	14,400,000	14,400,000	14,400,000	12,124,406	10,105,983	9,193,317	2,682,268	-	111,327,446
		DAu (g/t)	-	-	0.56	0.67	0.54	0.50	0.49	0.49	0.45	0.42	0.49	0.54	-	0.51

		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
To Stockpile	Pit to Stockpile	**High Grade (Au>0.4g/t)**	772,719	1,762,808	-	-	-	-	-	-	-	-	-	-	-	2,535,527
		DAu (g/t)	0.65	0.72	0.82	0.97	0.82	0.74	0.78	0.71	0.68	0.69	0.74	0.77	-	0.70
		Low Grade (0.196<Au<0.4 g/t)	873,923	2,169,078	-	-	-	-	-	-	-	-	-	-	-	3,043,001
		DAu (g/t)	0.29	0.28	0.28	0.28	0.27	0.28	0.28	0.27	0.28	0.27	0.28	0.29	-	0.28
		TOTAL ORE TONNES	1,646,642	3,931,886	-	-	-	-	-	-	-	-	-	-	-	5,578,528
		DAu (g/t)	0.46	0.48	-	-	-	-	-	-	-	-	-	-	-	0.47
	Total	**TOTAL ORE TONNES**	1,646,642	3,931,886	-	-	-	-	-	-	-	-	-	-	-	5,578,528
		DAu (g/t)	0.46	0.48	-	-	-	-	-	-	-	-	-	-	-	0.47

	Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11
From Stockpile	**TOTAL ORE TONNES**	-	-	3,924,919	1,437,787	215,822	-	-	-	-	-	-	-	-
	DAu (g/t)	-	-	0.55	0.28	0.28	-	-	-	-	-	-	-	-
	High Grade (Au>0.4g/t)			2,535,527										
	DAu (g/t)	-	-	0.70	-	-	-	-	-	-	-	-	-	-
	Low Grade (0.196<Au<0.4 g/t)			1,389,392	1,437,787	215,822								
	DAu (g/t)	-	-	0.28	0.28	0.28	-	-	-	-	-	-	-	-
Stockpile Inventory	**TOTAL ORE TONNES**	1,646,642	5,578,528	1,653,609	215,822	-	-	-	-	-	-	-	-	-
	DAu (g/t)	0.46	0.47	0.28	0.28	-	-	-	-	-	-	-	-	-
	High Grade (Au>0.4g/t)	772,719	2,535,527	-	-	-	-	-	-	-	-	-	-	-
	DAu (g/t)	0.65	0.70	-	-	-	-	-	-	-	-	-	-	-
	Low Grade (0.196<Au<0.4 g/t)	873,923	3,043,001	1,653,609	215,822	-	-	-	-	-	-	-	-	-
	DAu (g/t)	0.29	0.28	0.28	0.28	-	-	-	-	-	-	-	-	-

Cut Summary

	Report File		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
Phase 1			**High Grade (Au>0.4g/t)**	280,403	1,130,010	2,511,621	4,898,759	1,430,518	-	-	-	-	-	-	-	-	10,251,311
			DAu (g/t)	0.72	0.78	0.90	1.11	1.04	-	-	-	-	-	-	-	-	1.004
			Au (g/t)	0.78	0.82	0.92	1.13	1.07	-	-	-	-	-	-	-	-	1.026
			Low Grade (0.196<Au<0.4 g/t)	352,372	1,351,035	2,245,646	2,226,651	562,199	-	-	-	-	-	-	-	-	6,737,903
			DAu (g/t)	0.28	0.28	0.27	0.28	0.27	-	-	-	-	-	-	-	-	0.28
			Au (g/t)	0.30	0.29	0.28	0.29	0.28	-	-	-	-	-	-	-	-	0.29
			TOTAL ORE TONNES	632,775	2,481,045	4,757,267	7,125,410	1,992,717	-	-	-	-	-	-	-	-	16,989,214
			OB (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
			AG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
			PAG (tonnes)	621,810	2,513,604	3,025,399	3,047,385	283,164	-	-	-	-	-	-	-	-	9,491,362
			NAG (tonnes)	1,348,788	1,617,402	1,224,757	1,462,945	633,725	-	-	-	-	-	-	-	-	6,287,617
			TOTAL WASTE TONNES	1,970,598	4,131,006	4,250,156	4,510,330	916,889	-	-	-	-	-	-	-	-	15,778,979
			Total Tonnes	2,603,373	6,612,051	9,007,423	11,635,740	2,909,606	-	-	-	-	-	-	-	-	32,768,193
			S.R.	3.1	1.7	0.9	0.6	0.5	-	-	-	-	-	-	-	-	0.9
Phase 2			**High Grade (Au>0.4g/t)**	-	-	-	-	761,708	2,892,210	3,216,288	3,646,133	2,129,483	-	-	-	-	12,645,822
			DAu (g/t)	-	-	-	-	0.65	0.65	0.72	0.70	0.61	-	-	-	-	0.674
			Au (g/t)	-	-	-	-	0.67	0.67	0.74	0.71	0.62	-	-	-	-	0.689
			Low Grade (0.196<Au<0.4 g/t)	-	-	2,346	9,384	860,798	2,768,757	3,568,384	3,991,534	1,540,952	-	-	-	-	12,742,155
			DAu (g/t)	-	-	0.22	0.22	0.27	0.29	0.29	0.28	0.30	-	-	-	-	0.25
			Au (g/t)	-	-	0.25	0.26	0.28	0.30	0.30	0.29	0.30	-	-	-	-	0.26
			Total Ore Tonnes	-	-	2,346	9,384	1,622,506	5,660,967	6,784,672	7,637,667	3,670,435	-	-	-	-	25,387,977
			OB (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
			AG (tonnes)	-	-	-	-	-	-	12,510	-	-	-	-	-	-	12,510
			PAG (tonnes)	-	-	1,604,393	7,074,776	12,112,490	11,938,049	9,601,946	5,048,661	768,614	-	-	-	-	48,148,929
			NAG (tonnes)	-	-	174	82,739	898,593	675,195	649,363	188,683	3,378	-	-	-	-	2,498,125
			TOTAL WASTE TONNES	-	-	1,604,567	7,157,515	13,011,083	12,613,244	10,263,819	5,237,344	771,992	-	-	-	-	50,659,564
			Total Tonnes	-	-	1,606,913	7,166,899	14,633,589	18,274,211	17,048,491	12,875,011	4,442,427	-	-	-	-	76,047,541
			S.R.	-	-	684.0	762.7	8.0	2.2	1.5	0.7	0.2	-	-	-	-	2.0
Phase 3			**High Grade (Au>0.4g/t)**	492,316	632,798	1,038,062	1,766,975	3,362,231	2,019,114	2,043,616	1,351,505	255,767	-	-	-	-	12,962,384
			DAu (g/t)	0.61	0.61	0.62	0.69	0.87	0.97	0.96	0.84	0.89	-	-	-	-	0.829
			Au (g/t)	0.62	0.64	0.64	0.72	0.90	1.02	1.01	0.89	0.96	-	-	-	-	0.866
			Low Grade (0.196<Au<0.4 g/t)	521,551	818,043	1,077,406	2,382,005	2,489,070	1,258,661	1,222,165	1,475,334	109,149	-	-	-	-	11,353,384
			DAu (g/t)	0.29	0.28	0.29	0.29	0.27	0.26	0.27	0.26	0.29	-	-	-	-	0.27
			Au (g/t)	0.31	0.30	0.30	0.30	0.28	0.28	0.28	0.27	0.31	-	-	-	-	0.28
			Total Ore Tonnes	1,013,867	1,450,841	2,115,468	4,148,980	5,851,301	3,277,775	3,265,781	2,826,839	364,916	-	-	-	-	24,315,768
			OB (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
			AG (tonnes)	-	-	-	-	3,128	-	-	-	-	-	-	-	-	3,128
			PAG (tonnes)	551,741	976,698	1,764,593	5,118,603	5,914,770	3,698,806	2,147,686	1,220,571	21,204	-	-	-	-	21,414,671
			NAG (tonnes)	1,565,699	2,482,642	1,564,249	2,592,262	1,901,044	2,589,883	2,080,852	1,339,101	326,073	-	-	-	-	16,441,804
			TOTAL WASTE TONNES	2,117,440	3,459,340	3,328,842	7,710,865	7,818,942	6,288,689	4,228,537	2,559,671	347,277	-	-	-	-	37,859,603
			Total Tonnes	3,131,307	4,910,181	5,444,310	11,859,845	13,670,243	9,566,464	7,494,318	5,386,510	712,193	-	-	-	-	62,175,371
			S.R.	2.1	2.4	1.6	1.9	1.3	1.9	1.3	0.9	1.0	-	-	-	-	1.6
Phase 4			**High Grade (Au>0.4g/t)**	-	-	-	613,798	1,281,177	1,910,690	963,218	1,942,495	2,708,286	3,726,495	4,223,924	1,396,148	-	18,766,230
			DAu (g/t)	-	-	-	0.60	0.56	0.62	0.58	0.66	0.71	0.69	0.74	0.77	-	0.68
			Au (g/t)	-	-	-	0.63	0.58	0.63	0.61	0.68	0.74	0.71	0.76	0.80	-	0.71
			Low Grade (0.196<Au<0.4 g/t)	-	-	-	1,064,641	3,436,477	3,550,568	3,386,329	1,992,999	5,380,769	6,379,489	4,969,394	1,286,120	-	31,446,785
			DAu (g/t)	-	-	-	0.27	0.28	0.28	0.27	0.27	0.27	0.27	0.28	0.29	-	0.27
			Au (g/t)	-	-	-	0.28	0.29	0.29	0.28	0.29	0.28	0.28	0.29	0.30	-	0.29
			Total Ore Tonnes	-	-	-	1,678,439	4,717,654	5,461,258	4,349,547	3,935,494	8,089,055	10,105,983	9,193,317	2,682,268	-	50,213,015
			OB (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
			AG (tonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
			PAG (tonnes)	-	-	4,089,408	11,163,395	8,717,259	7,153,416	11,057,290	8,917,762	11,673,161	12,928,584	9,075,599	1,489,597	-	86,265,470
			NAG (tonnes)	-	-	-	354,233	2,280,574	5,012,230	6,132,526	4,339,969	8,210,312	8,856,955	3,839,632	536,116	-	39,562,545
			TOTAL WASTE TONNES	-	-	4,089,408	11,517,628	10,997,833	12,165,645	17,189,816	13,257,731	19,883,473	21,785,538	12,915,230	2,025,713	-	125,828,015
			Total Tonnes	-	-	4,089,408	13,196,067	15,715,487	17,626,903	21,539,363	17,193,225	27,972,528	31,891,521	22,108,547	4,707,981	-	176,041,030
			S.R.	-	-	-	6.9	2.3	2.2	4.0	3.4	2.5	2.2	1.4	0.8	-	2.5

Waste Dump Allocation

		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Open Pit Waste	**TOTAL WASTE TONNES**	**230,126,161**	4,088,038	7,590,346	13,272,973	30,896,338	32,744,747	31,067,578	31,682,172	21,054,746	21,002,742	21,785,538	12,915,230	2,025,713	230,126,161
	AG (tonnes)	**15,638**	-	-	-	-	3,128	-	12,510	-	-	-	-	-	15,638
	PAG (tonnes)	**165,320,432**	1,173,551	3,490,302	10,483,793	26,404,159	27,027,683	22,790,271	22,806,922	15,186,994	12,462,979	12,928,584	9,075,599	1,489,597	165,320,432
	NAG (tonnes)	**64,790,091**	2,914,487	4,100,044	2,789,180	4,492,179	5,713,936	8,277,308	8,862,741	5,867,753	8,539,763	8,856,955	3,839,632	536,116	64,790,091

Bank Volume (m3) — Rock Specific Gravity (t/m3): 2.78

		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Open Pit Waste	**TOTAL WASTE TONNES**	**82,779,195**	1,470,517	2,730,340	4,774,451	11,113,791	11,778,686	11,175,388	11,396,465	7,573,650	7,554,943	7,836,524	4,645,766	728,674	82,779,195
	AG (m3)	**5,625**	-	-	-	-	1,125	-	4,500	-	-	-	-	-	5,625
	PAG (m3)	**59,467,781**	422,141	1,255,504	3,771,149	9,497,899	9,722,188	8,197,939	8,203,929	5,462,947	4,483,086	4,650,570	3,264,604	535,826	59,467,781
	NAG (m3)	**23,305,788**	1,048,377	1,474,836	1,003,302	1,615,892	2,055,372	2,977,449	3,188,036	2,110,702	3,071,857	3,185,955	1,381,162	192,847	23,305,788

Loose Volume (lm3) — Swell Factor: 30%

		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Open Pit Waste	**Total Waste Volume**	**107,612,953**	1,911,673	3,549,443	6,206,786	14,447,928	15,312,292	14,528,004	14,815,404	9,845,744	9,821,426	10,187,481	6,039,496	947,276	107,612,953
	AG (m3)	**7,313**	-	-	-	-	1,463	-	5,850	-	-	-	-	-	7,313
	PAG (m3)	**77,308,116**	548,783	1,632,156	4,902,493	12,347,269	12,638,845	10,657,321	10,665,107	7,101,831	5,828,012	6,045,740	4,243,985	696,574	77,308,116
	NAG (m3)	**30,297,524**	1,362,890	1,917,287	1,304,293	2,100,659	2,671,984	3,870,683	4,144,447	2,743,913	3,993,414	4,141,741	1,795,511	250,702	30,297,524

Check Volumes (lm3)

		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Open Pit Waste	**Total Waste Volume**	**107,612,953**	1,911,673	3,549,443	6,206,786	14,447,928	15,312,292	14,528,004	14,815,404	9,845,744	9,821,426	10,187,481	6,039,496	947,276	107,612,953
	AG (m3)	**7,313**	-	-	-	-	1,463	-	5,850	-	-	-	-	-	7,313
	PAG (m3)	**77,308,116**	548,783	1,632,156	4,902,493	12,347,269	12,638,845	10,657,321	10,665,107	7,101,831	5,828,012	6,045,740	4,243,985	696,574	77,308,116
	NAG (m3)	**30,297,524**	1,362,890	1,917,287	1,304,293	2,100,659	2,671,984	3,870,683	4,144,447	2,743,913	3,993,414	4,141,741	1,795,511	250,702	30,297,524

Difference (lm3)

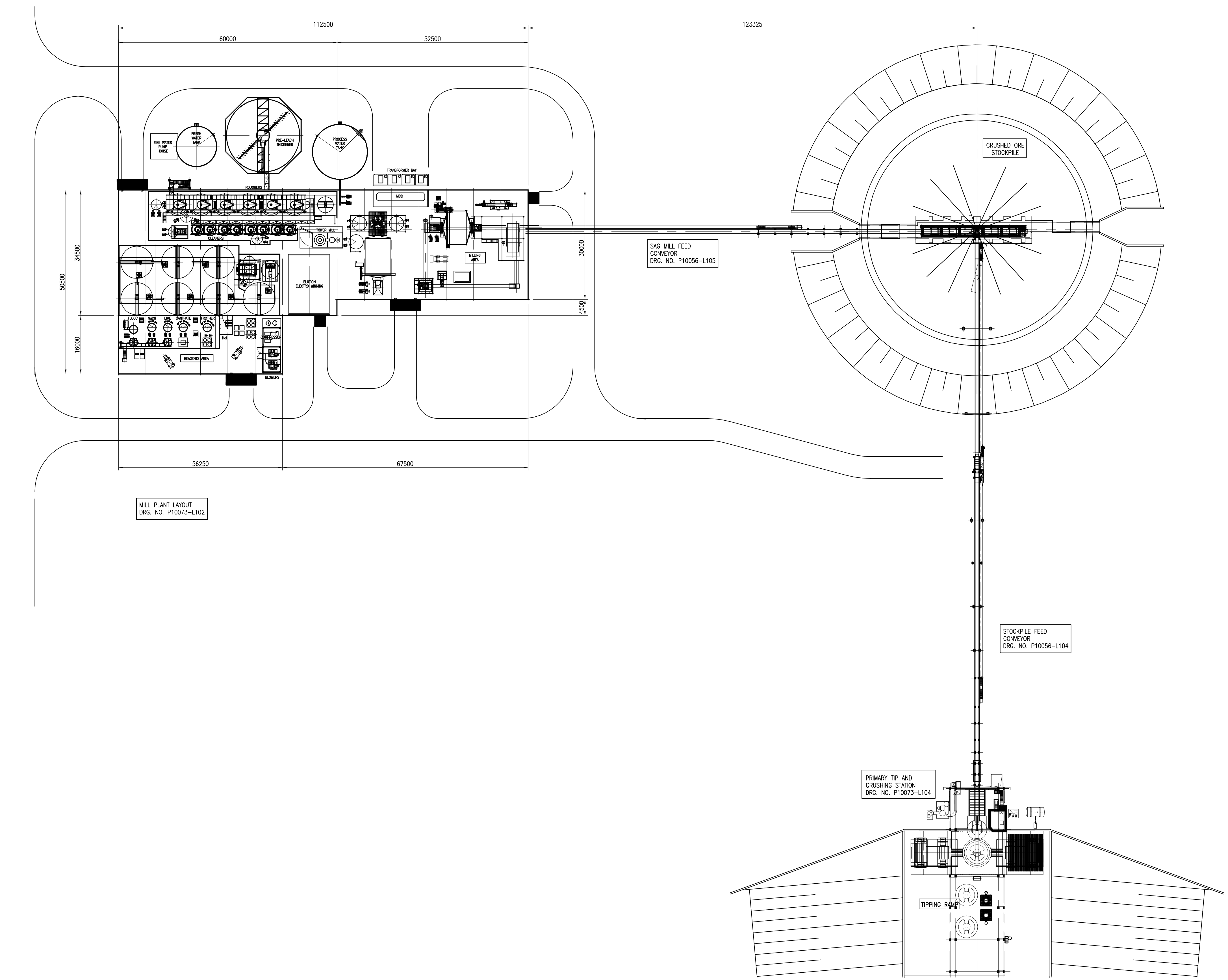
		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Open Pit Waste	**Total Waste Volume**	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-
	AG (m3)	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-
	PAG (m3)	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-
	NAG (m3)	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-

					Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
		Total			1,911,673	3,549,443	6,206,786	14,447,928	15,312,292	14,528,004	14,815,404	9,845,744	9,821,426	10,187,481	6,039,496	947,276
	West Dumps (includes Plant Dump)		Design (m3)	Actual (m3)												
West(no Plant dump)		Total	45,885,044	*66,628,579*	233,070	340,179	2,502,651	12,407,499	12,853,000	12,634,208	13,113,125	8,604,864	3,939,983	-	-	-
20,743,535		AG		-										-	-	-
Plant Dump		PAG		51,223,344			1,852,651	10,775,840	11,067,416	9,566,425	9,693,678	6,510,751	1,756,583			
45,885,044		NAG		15,405,235	233,070	340,179	650,000	1,631,659	1,785,584	3,067,783	3,419,447	2,094,113	2,183,400			
	East Dumps		Design (m3)	Actual (m3)												
		Total 1120 Level down	2,669,467	*3,009,646*	-	340,179	1,250,000	600,000	700,000	119,467	-	-	-	-	-	-
		AG		-												
		PAG		2,119,467			1,000,000	500,000	500,000	119,467						
		NAG		890,179		340,179	250,000	100,000	200,000							
		Total 1290 - 1120 Level	4,073,944	*4,647,193*	233,070	340,179	1,404,293	600,000	800,000	600,000	600,000	69,651	-	-	-	-
		AG		-												
		PAG		2,819,651			1,000,000	500,000	500,000	400,000	400,000	19,651				
		NAG		1,827,542	233,070	340,179	404,293	100,000	300,000	200,000	200,000	50,000				
	North Zone Backfill		Design (m3)	Actual (m3)												
		980 Level - Total		21,338,367									4,646,114	9,705,481	6,039,496	947,276
		AG		-									-	-	-	-
		PAG		14,486,299									3,500,000	6,045,740	4,243,985	696,574
		NAG		6,852,068									1,146,114	3,659,741	1,795,511	250,702
	Tailings Dam Construction															
		Northwest - Upstream	Design - NAG	421,600	88,000	88,000	-	38,700	30,100	43,100	34,200	36,100	37,500	25,900	-	-
			Actual - NAG	421,600	88,000	88,000	-	38,700	30,100	43,100	34,200	36,100	37,500	25,900	-	
		Northwest - Downstream	Design - NAG	3,300,500	594,050	594,050	-	151,100	242,800	376,700	314,600	357,900	387,100	282,200	-	-
			Actual - NAG	3,300,500	594,050	594,050	-	151,100	242,800	376,700	314,600	357,900	387,100	282,200	-	
		South East - Upstream	Design - NAG	227,800	30,200	30,200		19,100	16,800	25,300	26,800	28,400	30,200	20,800		
			Actual - NAG	227,800	30,200	30,200	-	19,100	16,800	25,300	26,800	28,400	30,200	20,800		
		South East - Downstream	Design - NAG	1,372,600	184,500	184,500		60,100	96,700	157,800	149,400	177,400	209,100	153,100		
			Actual - NAG	1,372,600	184,500	184,500	-	60,100	96,700	157,800	149,400	177,400	209,100	153,100		
		Total Material	Design - NAG	5,322,500	896,750	896,750	-	269,000	386,400	602,900	525,000	599,800	663,900	482,000	-	-
			Actual - NAG	5,322,500	896,750	896,750	-	269,000	386,400	602,900	525,000	599,800	663,900	482,000	-	-
		Difference	Actual - Design		-	-	-	-	-	-	-	-	-	-	-	-
	Tailings Placement															
		AG + PAG in TMF	Actual AG	7,313	-	-	-	-	1,463	-	5,850	-	-	-	-	-
			Actual PAG	6,659,355	548,783	1,632,156	1,049,842	571,429	571,429	571,429	571,429	571,429	571,429			
			Total	6,666,668	548,783	1,632,156	1,049,842	571,429	572,892	571,429	577,279	571,429	571,429	-	-	-

			Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Tonnage Calculation																
		West/East Dumps		996,822	2,182,379	11,027,926	29,099,113	30,693,338	28,556,320	29,324,990	18,550,117	8,425,502	-	-	-	
		North Zone Backfill		-	-	-	-	-	-	-	-	9,935,536	20,754,798	12,915,230	2,025,713	
		Tailings Dam - NW		1,458,538	1,458,538	-	405,880	583,586	897,726	745,895	842,554	907,991	658,860	-	-	
		Tailings Dam - SE		459,128	459,128	-	169,366	242,715	391,552	376,797	440,095	511,734	371,878	-	-	
		Tailings Placement		1,173,551	3,490,303	2,245,047	1,221,979	1,225,107	1,221,979	1,234,489	1,221,979	1,221,979	-	-	-	
		Total		4,088,039	7,590,347	13,272,973	30,896,338	32,744,747	31,067,578	31,682,172	21,054,745	21,002,742	21,785,536	12,915,230	2,025,713	
Tonnage Mined				Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
	Phase 1															
		AG		-	-	-	-	-	-	-	-	-	-	-	-	
		PAG		621,810	2,513,604	3,025,399	3,047,385	283,164	-	-	-	-	-	-	-	
		NAG		1,348,788	1,617,402	1,224,757	1,462,945	633,725	-	-	-	-	-	-	-	
		Total		1,970,598	4,131,006	4,250,156	4,510,330	916,889	-	-	-	-	-	-	-	
	Phase 2															
		AG		-	-	-	-	-	-	12,510	-	-	-	-	-	
		PAG		-	-	1,604,393	7,074,776	12,112,490	11,938,049	9,601,946	5,048,661	768,614	-	-	-	
		NAG		-	-	174	82,739	898,593	675,195	649,363	188,683	3,378	-	-	-	
		Total		-	-	1,604,567	7,157,515	13,011,083	12,613,244	10,263,819	5,237,344	771,992	-	-	-	
	Phase 3															
		AG		-	-	-	-	3,128	-	-	-	-	-	-	-	
		PAG		551,741	976,698	1,764,593	5,118,603	5,914,770	3,698,806	2,147,686	1,220,571	21,204	-	-	-	
		NAG		1,565,699	2,482,642	1,564,249	2,592,262	1,901,044	2,589,883	2,080,852	1,339,101	326,073	-	-	-	
		Total		2,117,440	3,459,340	3,328,842	7,710,865	7,818,942	6,288,689	4,228,537	2,559,671	347,277	-	-	-	
	Phase 4															
		AG		-	-	-	-	-	-	-	-	-	-	-	-	
		PAG		-	-	4,089,408	11,163,395	8,717,259	7,153,416	11,057,290	8,917,762	11,673,161	12,928,584	9,075,599	1,489,597	
		NAG		-	-	-	354,233	2,280,574	5,012,230	6,132,526	4,339,969	8,210,312	8,856,955	3,839,632	536,116	
		Total		-	-	4,089,408	11,517,628	10,997,833	12,165,645	17,189,816	13,257,731	19,883,473	21,785,538	12,915,230	2,025,713	
Dump Tonnages by Phase				Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
		West/East Dumps		-	-	-	-	(1)	(1)	(1)	(1)	-	-	-	-	
		North Zone Backfill		-	-	-	-	-	-	-	-	-	(2)	-	-	
		Tailings Dam - NW		-	-	-	-	-	-	-	-	-	-	-	-	
		Tailings Dam - SE		-	-	-	-	-	-	-	-	-	-	-	-	
		Tailings Placement		-	1	-	-	-	-	-	-	-	-	-	-	
	Phase 1	Difference		-	(1)	-	-	-	-	-	-	-	-	-	-	
		West/East Dumps			1,617,403	2,005,109	4,510,330	916,889								
		North Zone Backfill														
		Tailings Dam - NW		889,660												
		Tailings Dam - SE		459,128												
		Tailings Placement		621,810	2,513,604	2,245,047	-	-	-	-	-	-	-	-	-	
		Total		1,970,598	4,131,007	4,250,156	4,510,330	916,889	-	-	-	-	-	-	-	
	Phase 2	Difference		-	-	-	-	-	-	-	-	-	-	-	-	
		West/East Dumps				1,604,567	7,157,515	13,011,083	12,613,244	10,263,819	5,237,344	771,992				
		North Zone Backfill														
		Tailings Dam - NW														
		Tailings Dam - SE														
		Tailings Placement		-	-	-	-	-	-	-	-	-	-	-	-	
		Total		-	-	1,604,567	7,157,515	13,011,083	12,613,244	10,263,819	5,237,344	771,992	-	-	-	
	Phase 3	Difference		(1)	-	-	-	-	-	-	-	-	-	-	-	
		West/East Dumps		996,822	564,976	3,328,842	5,913,640	5,767,534	6,288,689	4,228,537	2,559,671	347,277				
		North Zone Backfill														
		Tailings Dam - NW		568,878	1,458,538		405,880	583,586								
		Tailings Dam - SE			459,128		169,366	242,715								
		Tailings Placement		551,741	976,698	-	1,221,979	1,225,107	-	-	-	-	-	-	-	
		Total		2,117,441	3,459,340	3,328,842	7,710,865	7,818,942	6,288,689	4,228,537	2,559,671	347,277	-	-	-	
	Phase 4	Difference		-	-	-	-	(0)	0	-	-	-	-	-	-	
		West/East Dumps				4,089,408	11,517,628	10,997,833	9,654,388	14,832,635	10,753,103	7,306,233				
		North Zone Backfill										9,935,536	20,754,800	12,915,230	2,025,713	
		Tailings Dam - NW							897,726	745,895	842,554	907,991	658,860			
		Tailings Dam - SE							391,552	376,797	440,095	511,734	371,878			
		Tailings Placement				-	-	-	1,221,979	1,234,489	1,221,979	1,221,979	-	-	-	
		Total		-	-	4,089,408	11,517,628	10,997,833	12,165,645	17,189,816	13,257,731	19,883,473	21,785,538	12,915,230	2,025,713	

APPENDIX E

Drawings



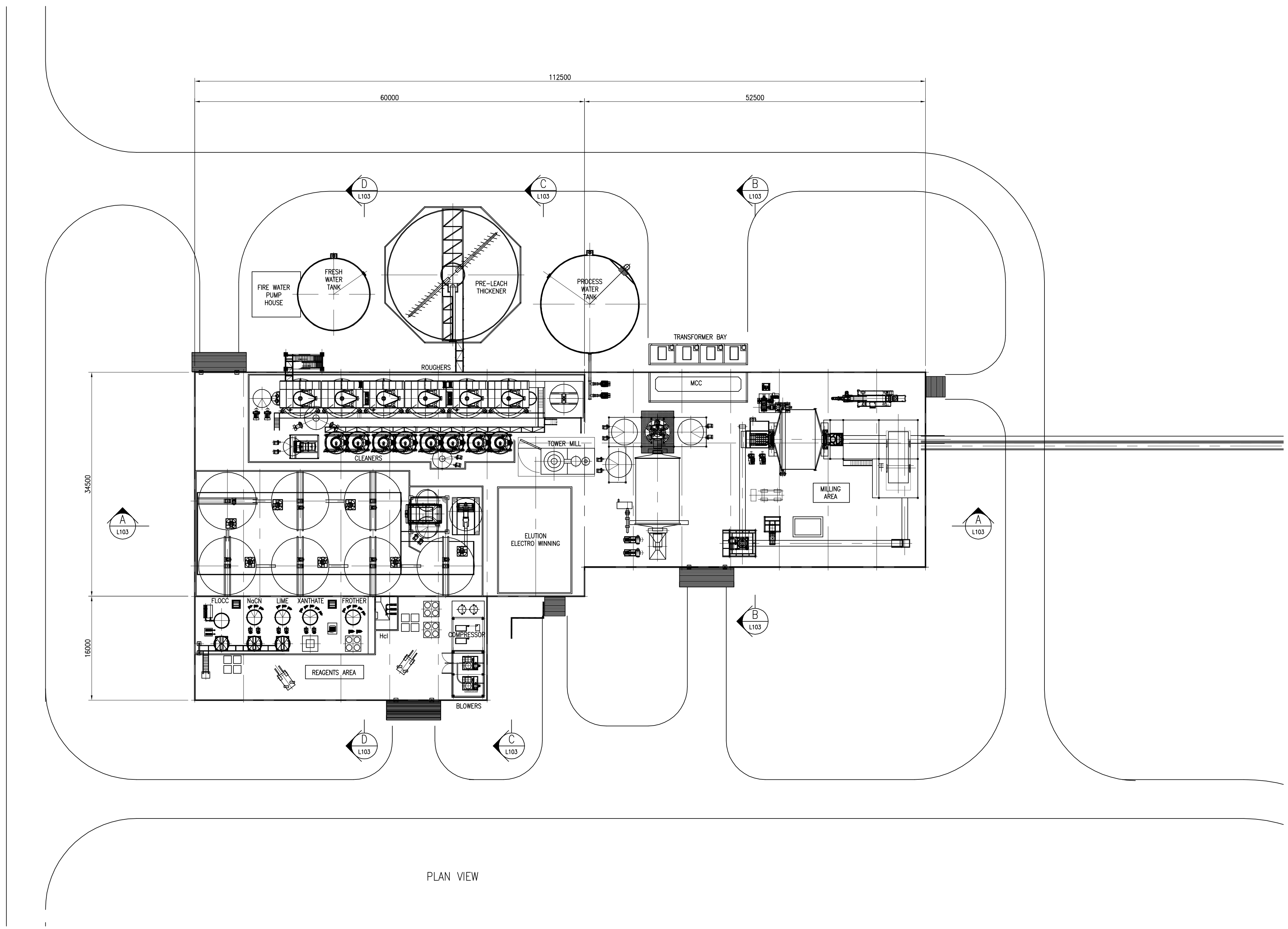
DRAWING NO.	DESCRIPTION	DRAWING NO.	DESCRIPTION	REV.	BY	DATE	REVISIONS	CHK.	DATE	S.L.	DATE
		P10073-L105	SAG MILL FEED CONVEYOR – PROPOSED LAYOUT								
		P10073-L104	STOCKPILE FEED CONVEYOR – PROPOSED LAYOUT								
		P10073-L103	PROPOSED PLANT LAYOUT – SECTIONS								
		P10073-L102	PROPOSED PLANT LAYOUT – PLAN VIEW	A	CPJS	04.10.10	ISSUED FOR ESTIMATING PURPOSES				
REFERENCE DRAWINGS		REFERENCE DRAWINGS					REVISIONS				

	APPROVAL	SIGNATURE	DATE
CLIENT			
PROJ. MANAGER			
PROF. ENG.			
PROJECT ENG.			
PROCESS ENG.			
D.O. MANAGER			
SECT LEADER			
CHECKED			
DRAWN		CPJS	28.09.10

DRA AMERICAS
380 ARMOUR ROAD, SUITE 100
PETERBOROUGH, ONTARIO, CANADA
K9H7L7
TEL No +1 (705) 742 3721
FAX No +1 (705) 872 4345

TITLE SPANISH MOUNTAIN GOLD
MINE PLANT
PROPOSED BLOCK PLAN

	DWG NO.	REVISION	SCALE
A0	P10073-L101	A	1:400 U.O.S.



PLAN VIEW

DRAWING NO.	DESCRIPTION	DRAWING NO.	DESCRIPTION	REV.	BY	DATE	REVISIONS	CHK.	DATE	S.L.	DATE
		P10073–L101	MINE PLANT – PROPOSED BLOCK PLAN	A	CPJS	04.10.10	ISSUED FOR ESTIMATING PURPOSES				
REFERENCE DRAWINGS		REFERENCE DRAWINGS									

APPROVAL	SIGNATURE	DATE
CLIENT		
PROJ. MANAGER		
PROF. ENG.		
PROJECT ENG.		
PROCESS ENG.		
D.O. MANAGER		
SECT LEADER		
CHECKED		
DRAWN	CPJS	28.09.10

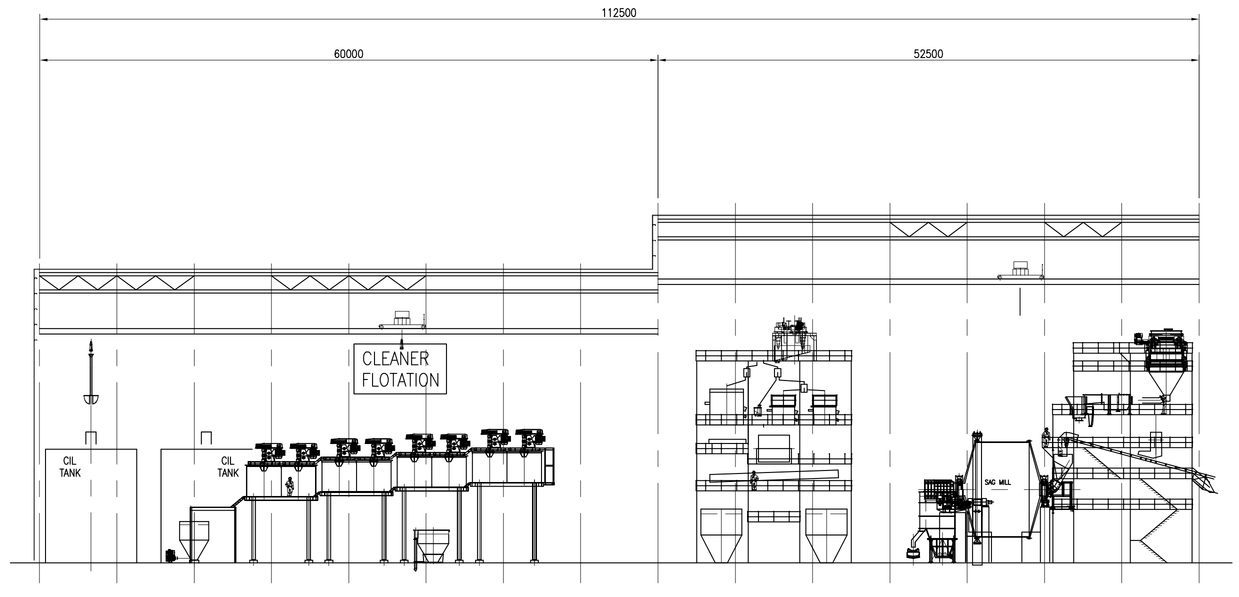
DRA AMERICAS

380 ARMOUR ROAD, SUITE 100
PETERBOROUGH, ONTARIO, CANADA
K9H7L7
TEL No +1 (705) 742 3721
FAX No +1 (705) 872 4345

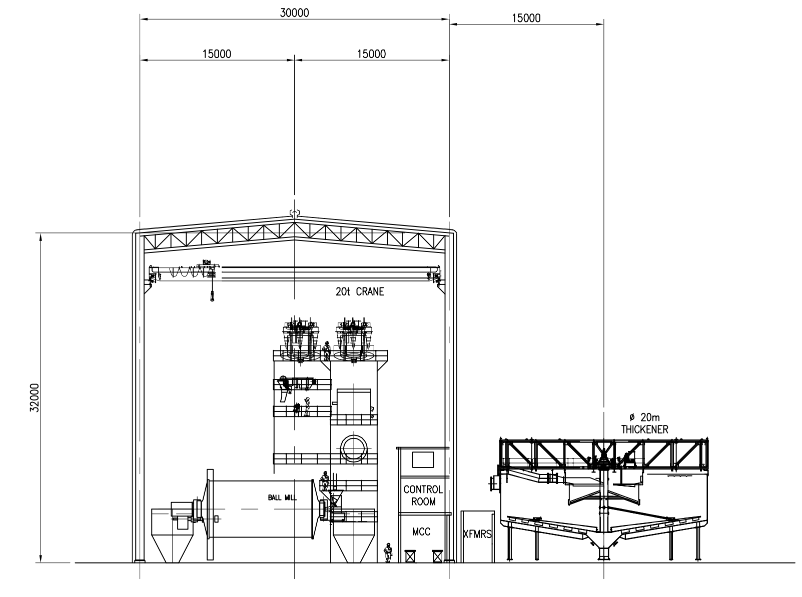
TITLE SPANISH MOUNTAIN GOLD
PROCESS PLANT
PROPOSED PLANT LAYOUT
PLAN VIEW

	DWG NO.	REVISION	SCALE
A0	P10073–L102	A	1:250 U.O.S.

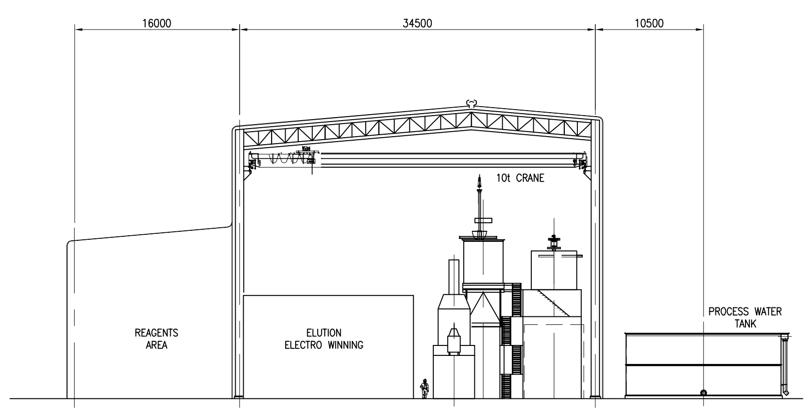
Z:\500 - Drawing Office\Projects\P10073 SPANISH MOUNTAIN GOLD\Layouts\P10073-L102~A.dwg, 2010/12/06 03:28:23 PM, Noel Holman, DRA Americas



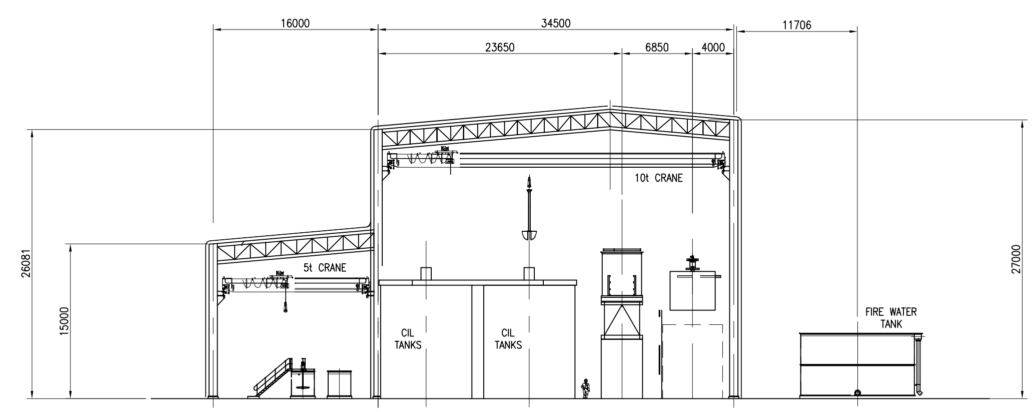
SECTION A L102



SECTION B L102



SECTION C L102



SECTION D L102

DRAWING NO.	DESCRIPTION	DRAWING NO.	DESCRIPTION	REV.	BY	DATE	REVISIONS	CHK.	DATE	S.L.	DATE
		P10073-L101	MINE PLANT - PROPOSED BLOCK PLAN	A	CPJS	04.10.10	ISSUED FOR ESTIMATING				
REFERENCE DRAWINGS		REFERENCE DRAWINGS					REVISIONS				

	APPROVAL	SIGNATURE	DATE
CLIENT			
PROJ. MANAGER			
PROF. ENG.			
PROJECT ENG.			
PROCESS ENG.			
D.O. MANAGER			
SECT LEADER			
CHECKED			
DRAWN		CPJS	28.09.10

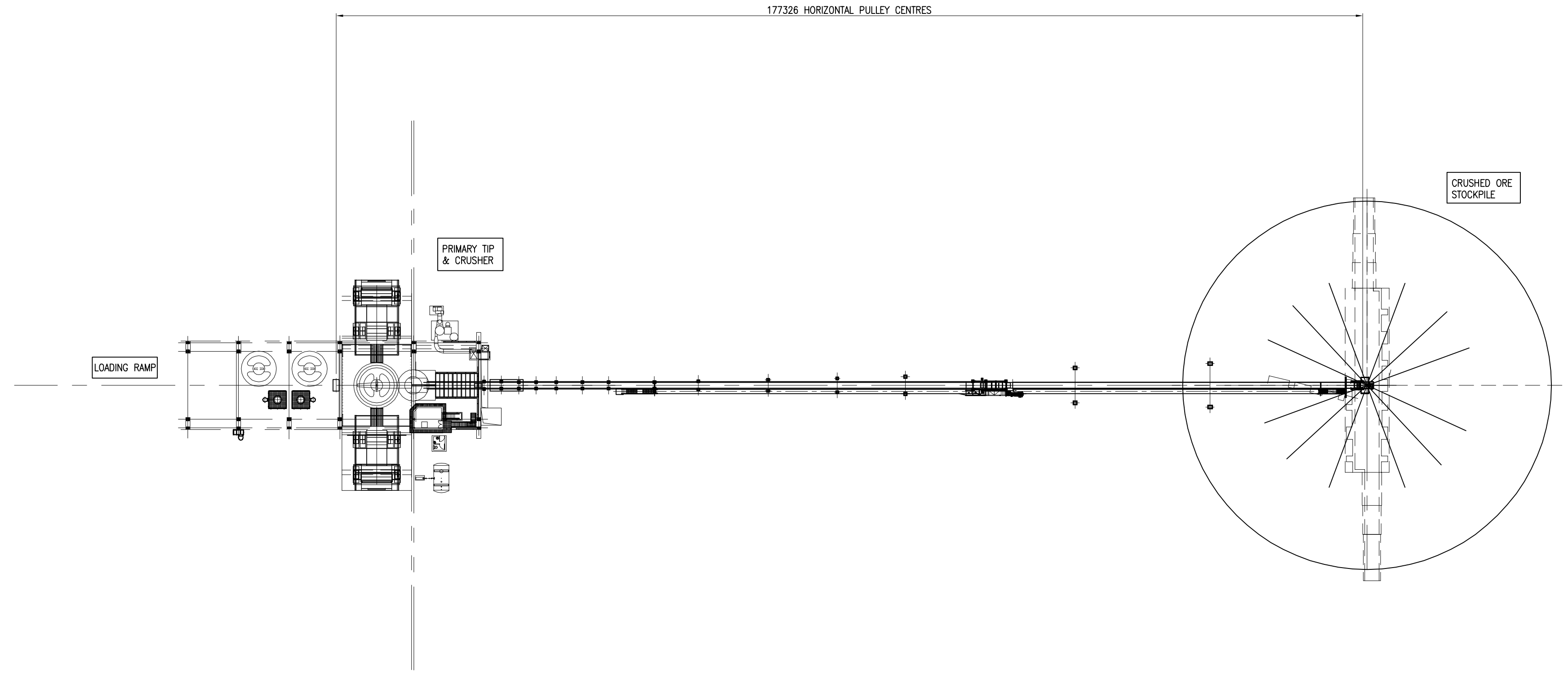
DRA AMERICAS

380 ARMOUR ROAD, SUITE 100
PETERBOROUGH, ONTARIO, CANADA
K9H7L7
TEL No +1 (705) 742 3721
FAX No +1 (705) 872 4345

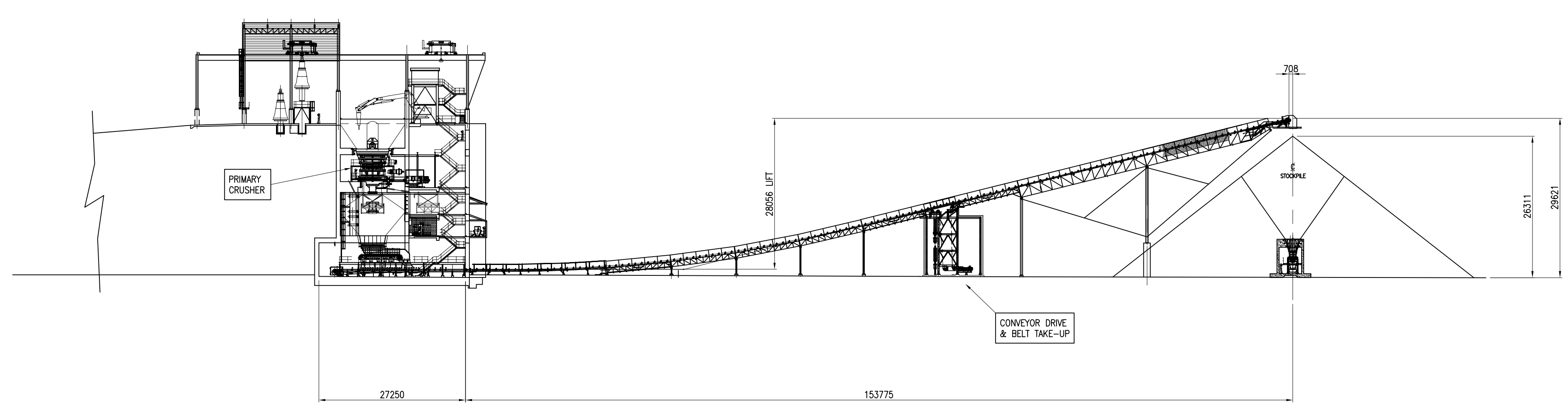
TITLE SPANISH MOUNTAIN GOLD
PROCESS PLANT
PROPOSED PLANT LAYOUT
SECTIONS

A0	DWG NO. P10073-L103	REVISION A	SCALE 1:250 U.O.S.

Z:\500 - Drawing Office\Projects\P10073 SPANISH MOUNTAIN GOLD\Layouts\P10073-L103~A.dwg, 2010/12/06 03:26:40 PM, Noel Holman, DRA Americas



PLAN ON STOCKPILE FEED CONVEYOR



ELEVATION ON STOCKPILE FEED CONVEYOR

DRAWING NO.	DESCRIPTION	DRAWING NO.	DESCRIPTION	REV.	BY	DATE	REVISIONS	CHK.	DATE	S.L.	DATE
		P10073-L101	MINE PLANT - PROPOSED BLOCK PLAN	A	CPJS	29.09.10	ISSUED FOR ESTIMATING				
				A	CPJS	29.09.10					
REFERENCE DRAWINGS		REFERENCE DRAWINGS									

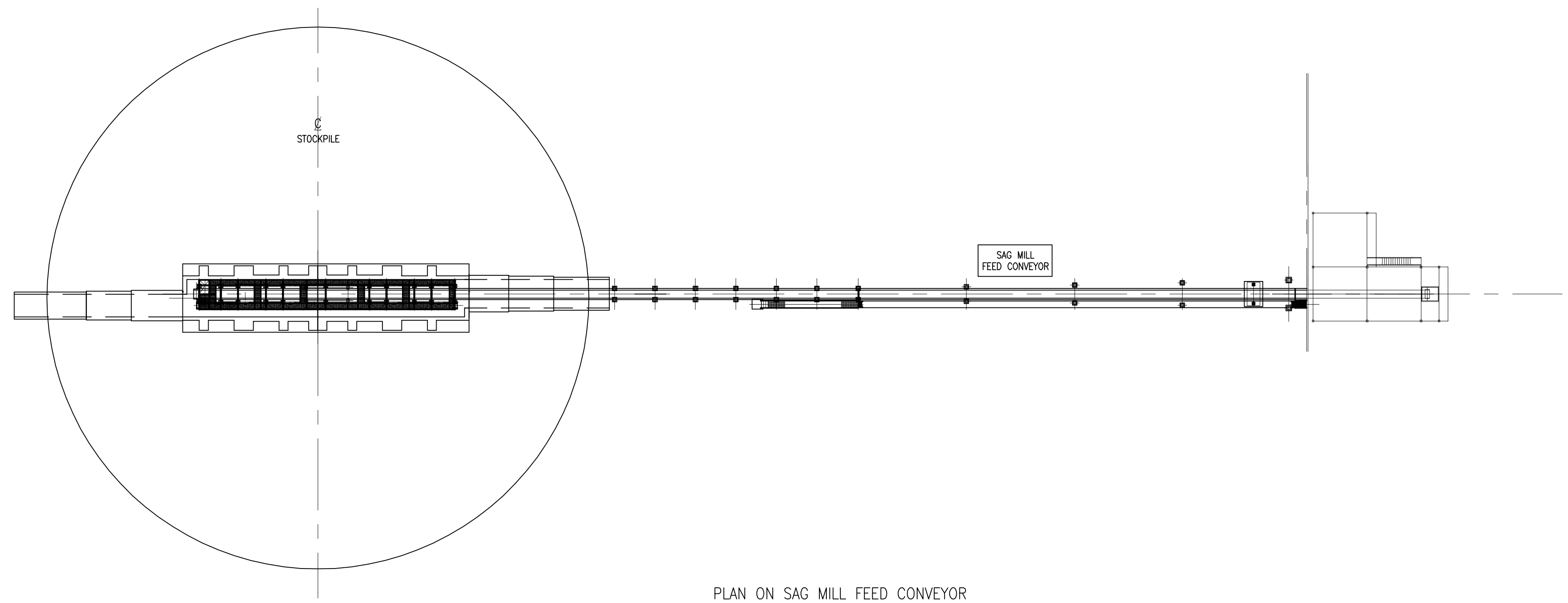
	SIGNATURE	DATE
CLIENT		
PROJ. MANAGER		
PROF. ENG.		
PROJECT ENG.		
PROCESS ENG.		
D.O. MANAGER		
SECT LEADER		
CHECKED		
DRAWN	CPJS	29.09.10
APPROVAL		

DRA AMERICAS

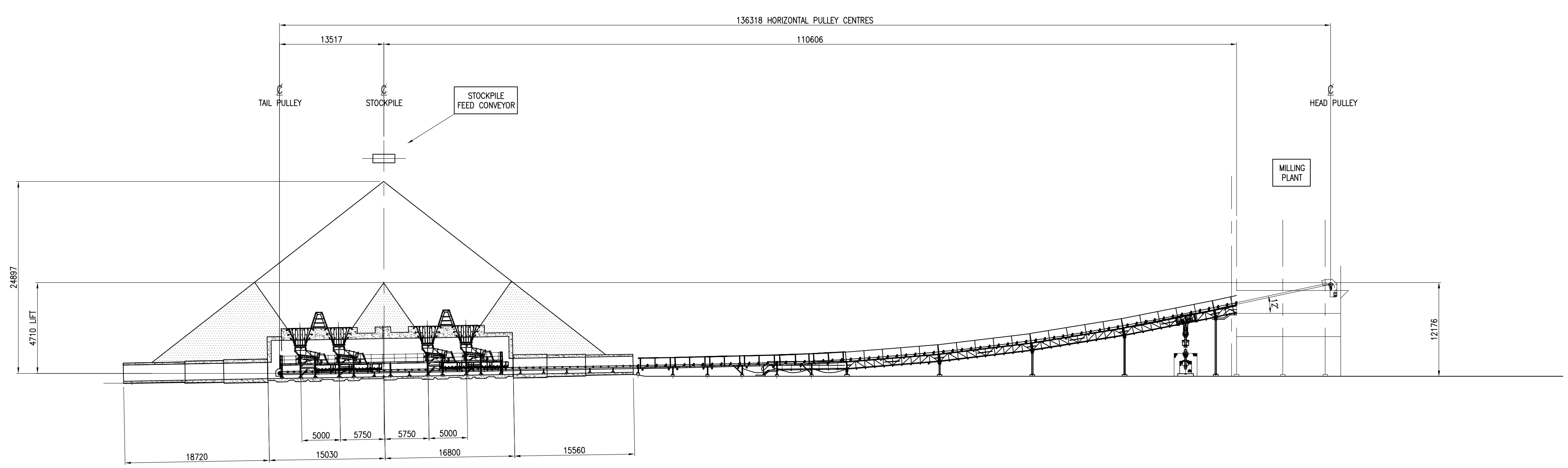
380 ARMOUR ROAD, SUITE 100
PETERBOROUGH, ONTARIO, CANADA
K9H7L7
TEL No +1 (705) 742 3721
FAX No +1 (705) 872 4345

TITLE SPANISH MOUNTAIN GOLD
PROCESS PLANT
STOCKPILE FEED CONVEYOR
PROPOSED LAYOUT

A0 | DWG NO. P10073-L104 | REVISION A | SCALE 1:150 U.O.S.



PLAN ON SAG MILL FEED CONVEYOR



ELEVATION ON SAG MILL FEED CONVEYOR

DRAWING NO.	DESCRIPTION	DRAWING NO.	DESCRIPTION	REV.	BY	DATE	REVISIONS	CHK.	DATE	S.L.	DATE
		P10073-L101	MINE PLANT - PROPOSED BLOCK PLAN	A	CPJS	29.09.10	ISSUED FOR ESTIMATING				
REFERENCE DRAWINGS		REFERENCE DRAWINGS									

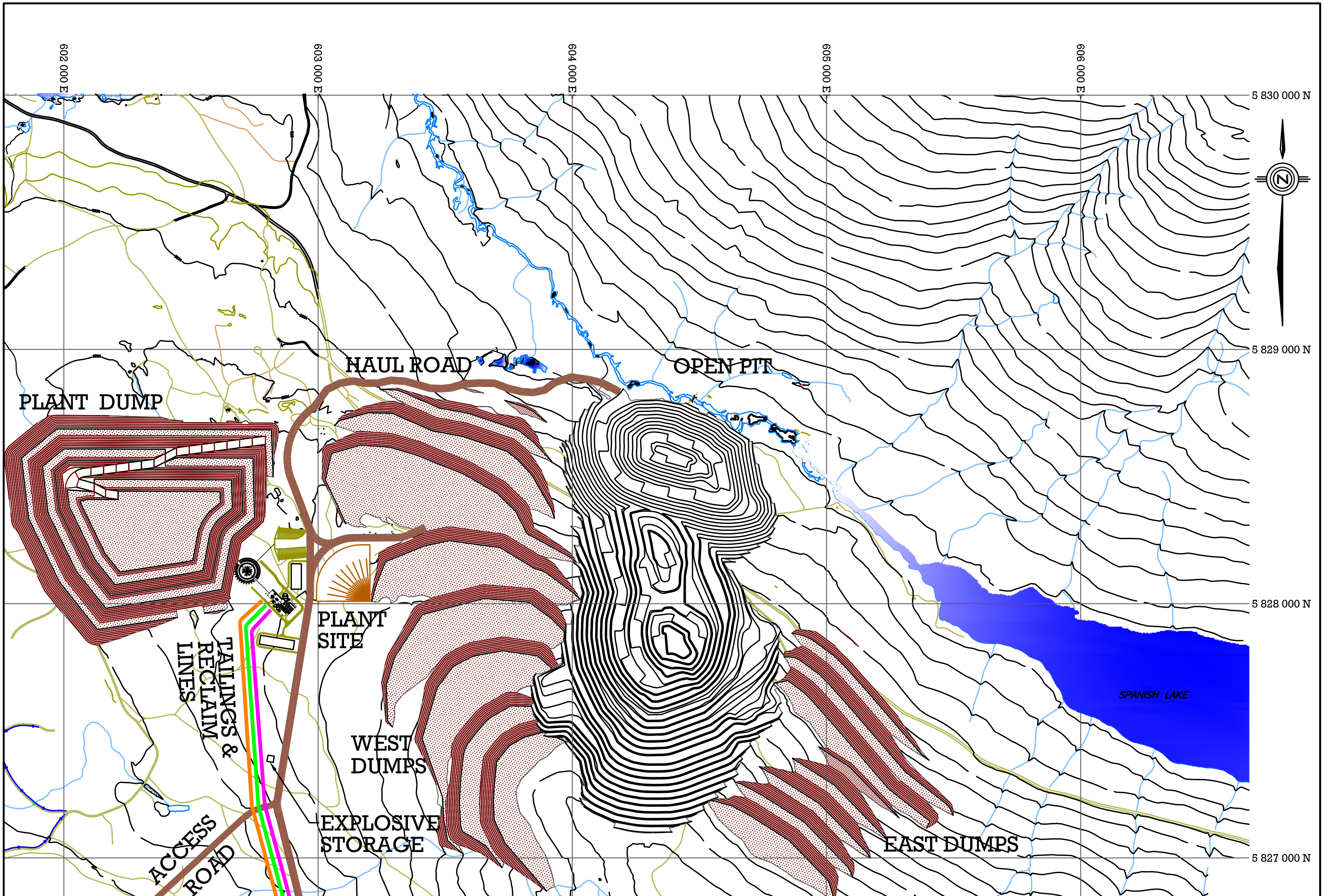
CLIENT		
PROJ. MANAGER		
PROF. ENG.		
PROJECT ENG.		
PROCESS ENG.		
D.O. MANAGER		
SECT LEADER		
CHECKED		
DRAWN	CPJS	29.09.10
APPROVAL	SIGNATURE	DATE

DRA AMERICAS

380 ARMOUR ROAD, SUITE 100
PETERBOROUGH, ONTARIO, CANADA
K9H7L7
TEL No +1 (705) 742 3721
FAX No +1 (705) 872 4345

TITLE SPANISH MOUNTAIN GOLD
PROCESS PLANT
SAG MILL FEED CONVEYOR
PROPOSED LAYOUT

	DWG NO.	REVISION	SCALE
A0	P10073–L105	A	1:150 U.O.S.



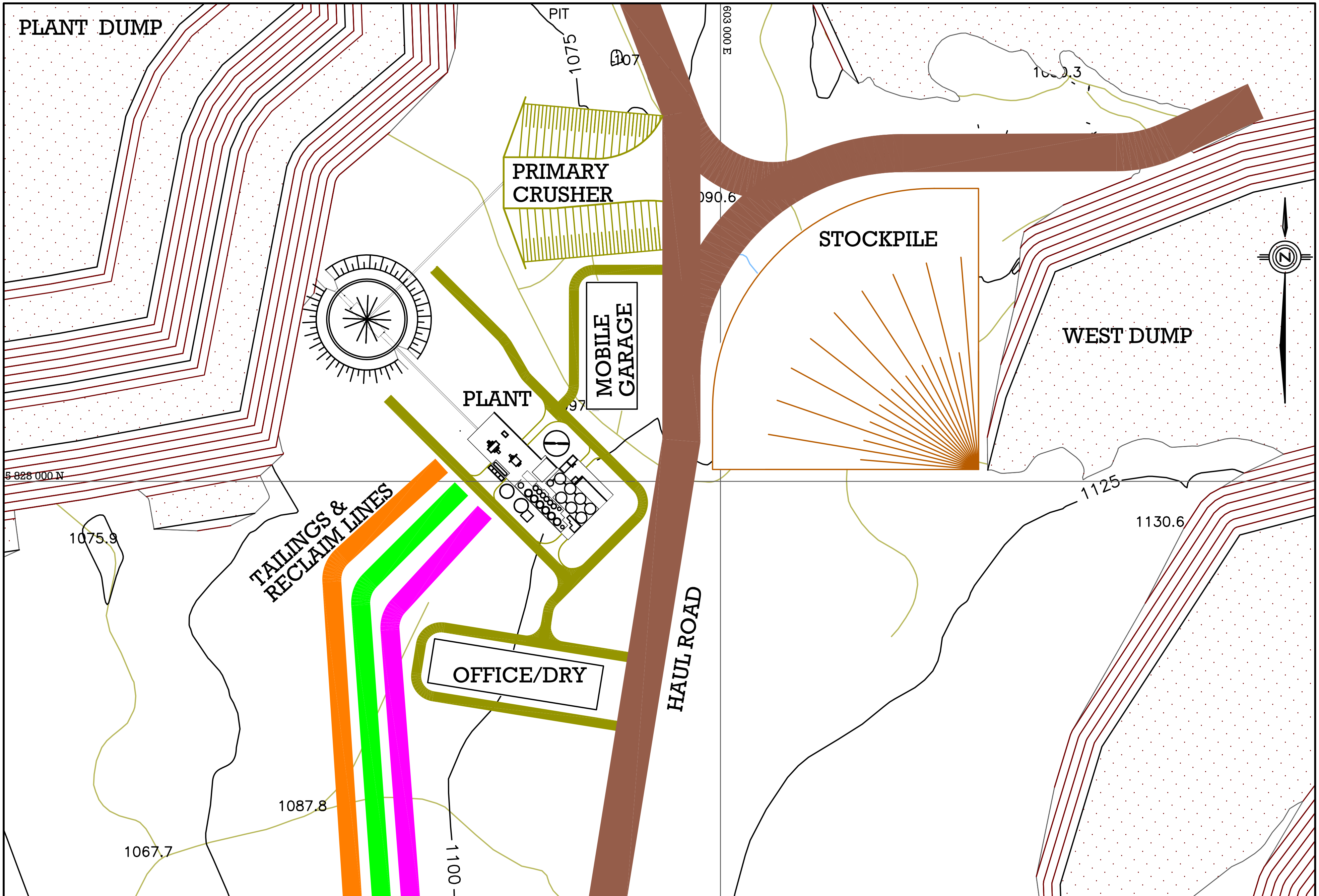
Spanish Mountain Gold Ltd.

SPANISH MOUNTAIN PROJECT

0 500 1000
(METERS)

Spanish Mountain Mine Site

AUTHOR:	S.B.D.	DATE:	DECEMBER 2010

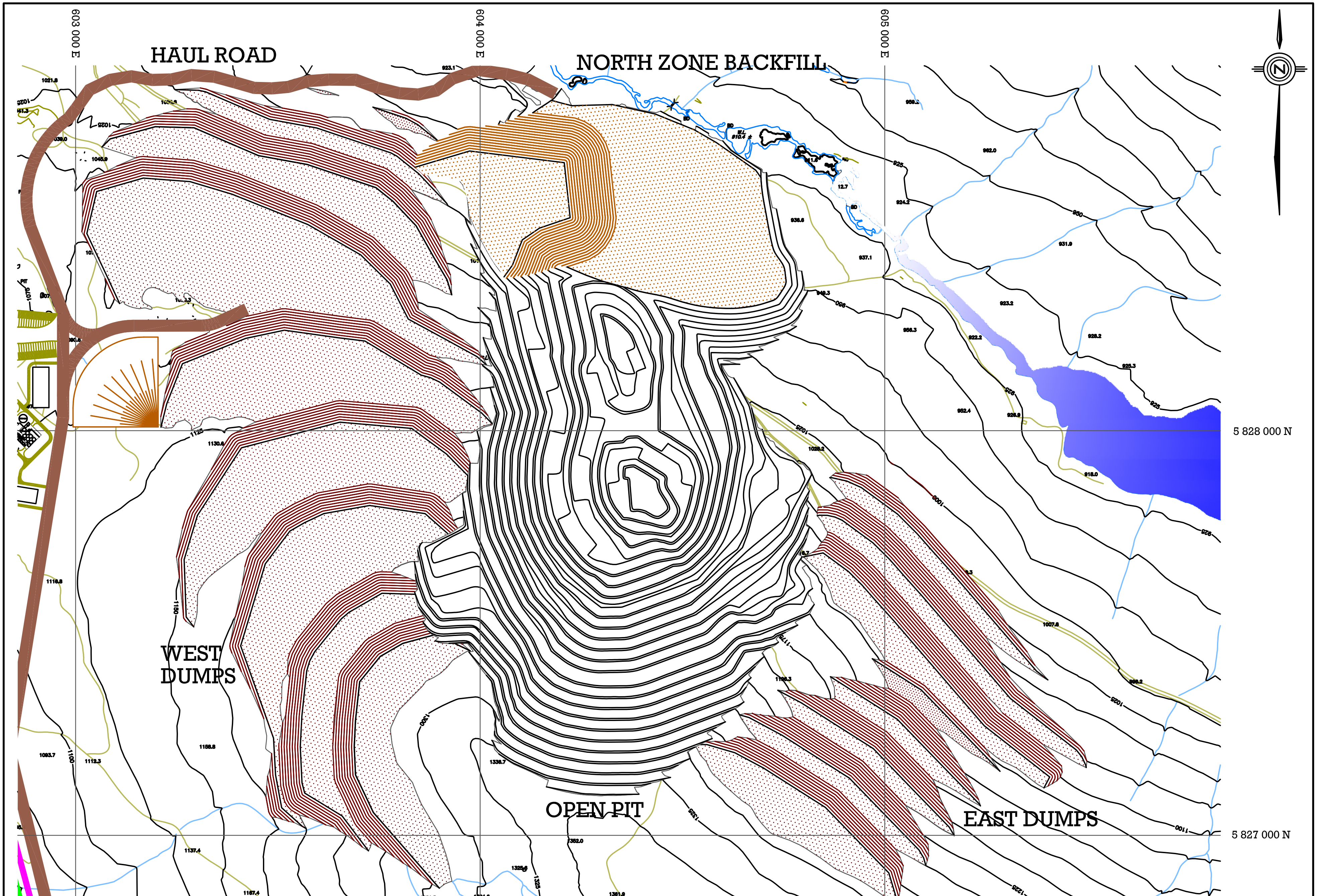


Spanish Mountain Gold Ltd.

SPANISH MOUNTAIN PROJECT

Spanish Mountain Mill Site

AUTHOR:	S.B.D.	DATE:	DECEMBER 2010



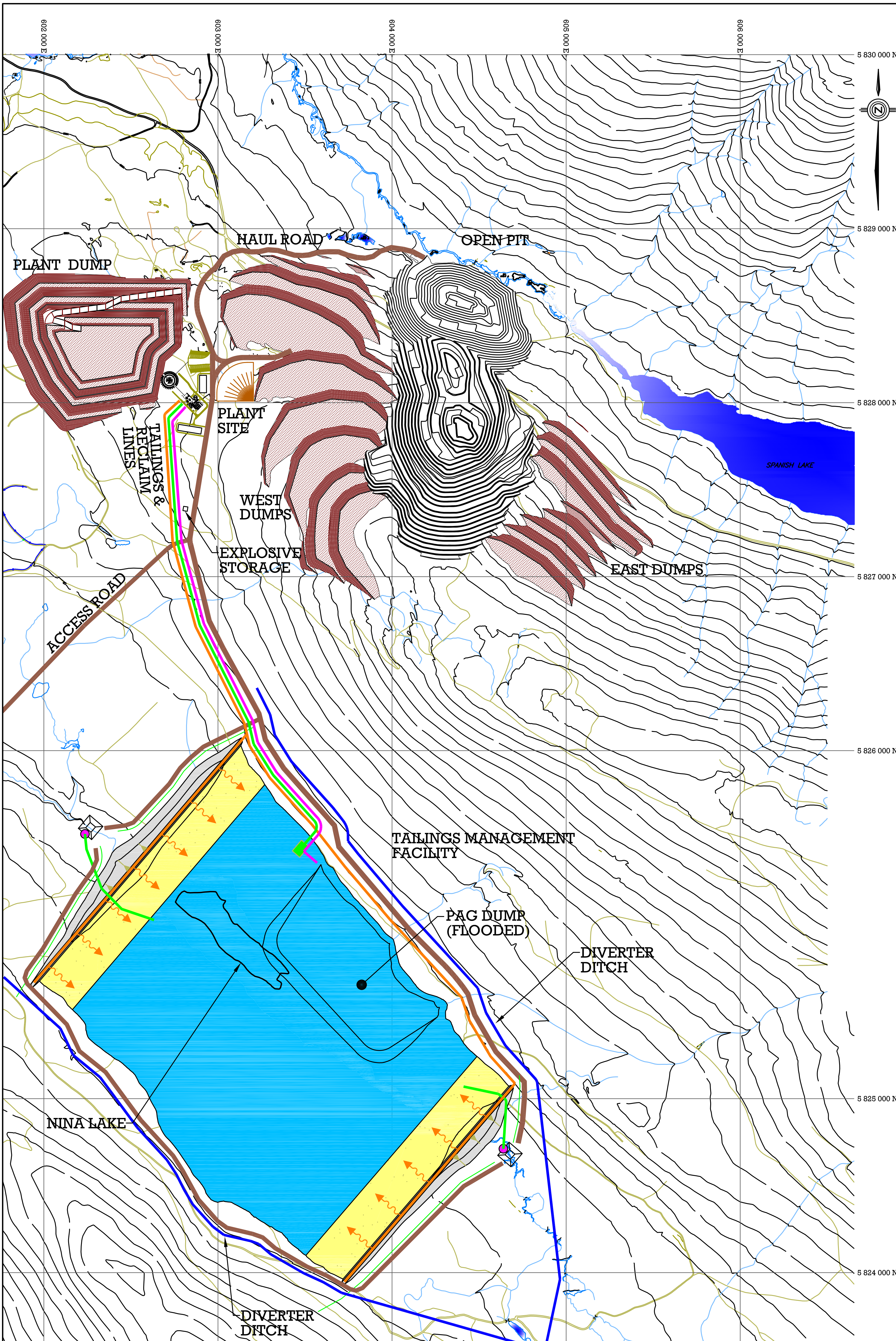
Spanish Mountain Gold Ltd.

SPANISH MOUNTAIN PROJECT

0 250 500
(METERS)

Final Waste Dump Configuration with North Zone Backfill

AUTHOR:	S.B.D.	DATE:	DECEMBER 2010



Spanish Mountain Gold Ltd.

SPANISH MOUNTAIN PROJECT

0 500 1000 (METERS)

Spanish Mountain Site

AUTHOR: S.B.D. DATE: DECEMBER 2010

APPENDIX F

Capital and Operating Costs

Capital Costs - Final 40K Option

	Unit Cost $ US	Unit Life	Total Capital Cost $Cdn	Units Required											
				Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Underground Capital															
Equipment Capital			$ -												
Development Capital			$ -												
Ventilation, OP, WP Capital			$ -												
Underground Mining Capital			**$ -**												
Processing Capital															
Plant	$ 212,900,000		$ 212,900,000	0.3	0.5	0.2									
			$ -												
			$ -												
Sustaining Capital (@1% of initial capital)	$ 2,129,000		$ 2,129,000					0.1	0.1	0.2	0.2	0.2	0.1	0.1	
Subtotal Processing Capital			$ 215,029,000												
Infrastructure Capital															
Power - Power line upgrade	$ 8,000,000		$ 8,000,000	1											
Power - Electrical Substations	$ 13,600,000		$ 13,600,000	1											
Power - Pit Powerlines	$ 400,000		$ 400,000		1										
Explosives Storage Area	$ 300,000		$ 300,000	1											
Haul Road Construction	$ 3,500,000		$ 3,500,000	0.5	0.5										
Fuel Storage	$ 300,000		$ 300,000	1											
Shop and Garage	$ 8,650,000		$ 8,650,000	0.75	0.25										
Fresh Water and Pumping System	$ 1,800,000		$ 1,800,000	0.3	0.5	0.2									
			$ -												
Owners Cost	$ 8,000,000		$ 8,000,000	0.5	0.5										
Mobile Equipment	$ 500,000		$ 500,000	0.5	0.5										
Communications	$ 150,000		$ 150,000	1											
Office	$ 3,500,000		$ 3,500,000		0.75	0.25									
Access Road to Plant	$ 1,750,000		$ 1,750,000	1											
Construction Camp			$ -												
			$ -												
Tailings Management Area - Initial	$ 27,864,000		$ 27,864,000	0.37	0.63										
Tailings Management Area - Sustaining	$ 9,134,000		$ 9,134,000			0.00	0.10	0.14	0.13	0.17	0.12	0.18	0.09	0.06	-
			$ -												
Subtotal Infrastructure Capital	$ 87,448,000		$ 87,448,000												
Environmental Costs															
Fisheries Compensation	$ 10,000,000		$ 10,000,000	1.000											
Interest on Financial Assurance	$ 8,500,000		$ 8,500,000	1.000											
			$ -												
Subtotal Environmental Costs			$ 18,500,000												
Indirect Costs															
Open Pit Mining Indirects	$ -		$ -	0.5	0.5										
Underground Mining Indirects	$ -		$ -												
Processing Indirects	$ 46,224,784		$ 46,224,784	0.3	0.5	0.2									
Infrastructure Indirects	$ 24,149,184		$ 24,149,184	0.47	0.38	0.01	0.02	0.02	0.02	0.02	0.02	0.03	0.01	0.01	-
Environmental Indirects	$ -		$ -	1											
Subtotal Indirect Costs			$ 70,373,968												
Contingency															
Open Pit Contingency	$ -		$ -	0.5	0.5										
Underground Contingency	$ -		$ -												
Processing Contingency	$ 54,241,065		$ 54,241,065	0.3	0.5	0.2									
Infrastructure Contingency	$ 15,852,503		$ 15,852,503	0.50	0.35	0.01	0.02	0.02	0.02	0.02	0.02	0.02	0.01	0.01	-
Environmental Contingency	$ 2,000,000		$ 2,000,000	1											
Subtotal Contingency			$ 72,093,568												

TMF Capital Cost - Unadjusted

			Stage	1		2	3	4	5	6	7	8	0		
Item Number	Description	Units	Unit Cost	Year PreProduction Cost	Year 1 Cost	Year 2 Cost	Year 3 Cost	Year 4 Cost	Year 5 Cost	Year 6 Cost	Year 7 Cost	Year 8 Cost	Year 9 Cost	Year 10 Cost	Total
1.1	**HEAVY CIVIL**														
A	**Site Preparation**														
	Logging	ha	$1,500	$435,000	$0	$66,000	$49,500	$67,500	$45,000	$43,500	$25,500	$18,000	$22,500	$0	$772,500
	Service roads	km	$30,000	$172,500	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$172,500
	Pipeline corridor construction	km	$30,000	$165,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$165,000
	Construction dewatering (including cofferdams) - NW Embankment	LS	$400,000	$400,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$400,000
	Construction dewatering (including cofferdams) - SE Embankment	LS	$400,000	$400,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$400,000
	Sediment and erosion control BMPs	ha	$1,250	$362,500	$0	$55,000	$41,250	$56,250	$37,500	$36,250	$21,250	$15,000	$18,750	$0	$643,750
			Sub-Total	**$1,935,000**	**$0**	**$121,000**	**$90,750**	**$123,750**	**$82,500**	**$79,750**	**$46,750**	**$33,000**	**$41,250**	**$0**	**$2,553,750**
B	**Waste Management Facility**														
	Embankment footprint foundation preparation - NW Embankment	m^2	$1.10	$206,800	$0	$5,830	$5,830	$5,830	$5,830	$5,830	$5,830	$5,830	$5,830	$0	$253,440
	Embankment footprint foundation preparation - SE Embankment	m^2	$1.10	$68,750	$0	$13,310	$13,310	$13,310	$13,310	$13,310	$13,310	$13,310	$13,310	$0	$175,230
	Foundation drains - NW Embankment	m	$115	$230,000	$0	$5,750	$5,750	$5,750	$5,750	$5,750	$5,750	$5,750	$5,750	$0	$276,000
	Foundation drains - SE Embankment	m	$115	$103,500	$0	$14,950	$14,950	$14,950	$14,950	$14,950	$14,950	$14,950	$14,950	$0	$223,100
	Embankment Filter Zone - NW Embankment	m^3	$19	$987,470	$0	$105,050	$82,130	$116,510	$93,590	$99,320	$101,230	$70,670	$0	$0	$1,655,970
	Embankment Filter Zone - SE Embankment	m^3	$19	$328,520	$0	$51,570	$45,840	$68,760	$72,580	$78,310	$84,040	$57,300	$0	$0	$786,920
	Embankment Transition Zone - NW Embankment	m^3	$16	$827,200	$0	$88,000	$68,800	$97,600	$78,400	$83,200	$84,800	$59,200	$0	$0	$1,387,200
	Embankment Transition Zone - SE Embankment	m^3	$16	$275,200	$0	$43,200	$38,400	$57,600	$60,800	$65,600	$70,400	$48,000	$0	$0	$659,200
	Embankment Core Zone - NW Embankment	m^3	$5.50	$2,751,100	$0	$266,750	$208,450	$299,200	$234,850	$251,900	$257,950	$178,750	$0	$0	$4,448,950
	Embankment Core Zone - SE Embankment	m^3	$5.50	$926,200	$0	$132,000	$116,600	$174,900	$184,800	$196,900	$210,100	$143,000	$0	$0	$2,084,500
	Upstream Embankment Shell Zone - NW Embankment	m^3	$0.40	$70,400	$0	$15,480	$12,040	$17,240	$13,680	$14,440	$15,000	$10,360	$0	$0	$168,640
	Upstream Embankment Shell Zone - SE Embankment	m^3	$0.40	$24,160	$0	$7,640	$6,720	$10,120	$10,720	$11,360	$12,080	$8,320	$0	$0	$91,120
	Downstream Embankment Shell Zone - NW Embankment	m^3	$0.40	$475,240	$0	$60,440	$97,120	$150,680	$125,840	$143,160	$154,840	$112,880	$0	$0	$1,320,200
	Downstream Embankment Shell Zone - SE Embankment	m^3	$0.40	$147,600	$0	$24,040	$38,680	$63,120	$59,760	$70,960	$83,640	$61,240	$0	$0	$549,040
	Seepage recovery and recycle systems	LS	$250,000	$500,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$500,000
			Sub-Total	**$7,922,140**	**$0**	**$834,010**	**$754,620**	**$1,095,570**	**$974,860**	**$1,054,990**	**$1,113,920**	**$789,560**	**$39,840**	**$0**	**$14,579,510**
C	**Diversion Ditches**														
	Diversion ditch extensions	m	$30	$75,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$75,000
	Boswell Lake overflow channel	LS	$500,000	$500,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$500,000
			Sub-Total	**$575,000**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$575,000**
			Heavy Civil Sub-Total	**$10,432,140**	**$0**	**$955,010**	**$845,370**	**$1,219,320**	**$1,057,360**	**$1,134,740**	**$1,160,670**	**$822,560**	**$81,090**	**$0**	**$17,708,260**
1.2	**ELECTRICAL**														
	Lighting	LS	$12,300	$12,300	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$12,300
	Junction boxes and transformers	LS	$280,000	$280,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$280,000
	MCC	LS	$122,000	$122,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$122,000
	Grounding	LS	$40,000	$40,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$40,000
	PLC	LS	$38,650	$38,650	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$38,650
	Distribution line	km	$120,000	$840,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$840,000
	Cable	LS	$520,000	$520,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$520,000
			Electrical Sub-Total	**$1,852,950**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$1,852,950**
1.3	**MECHANICAL**														
A	**Rougher System**														
	Rougher tailings pipeline	m	$800	$7,440,000	$0	$0	$240,000	$0	$240,000	$0	$240,000	$0	$240,000	$0	$8,400,000
	Offtake valves	ea	$100,000	$200,000	$0	$0	$100,000	$0	$100,000	$0	$100,000	$0	$100,000	$0	$600,000
	HDPE discharge pipe	m	$150	$18,750	$0	$0	$9,750	$0	$9,750	$0	$9,000	$0	$9,000	$0	$56,250
B	**Cleaner System**														
	Cleaner tailings pipeline	m	$150	$600,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$600,000
C	**Reclaim System**														
	Reclaim pipeline	m	$700	$2,730,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$2,730,000
	Reclaim barge including instrumentation/communication	ea	$4,100,000	$4,100,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$4,100,000
			Mechanical Sub-Total	**$15,088,750**	**$0**	**$0**	**$349,750**	**$0**	**$349,750**	**$0**	**$349,000**	**$0**	**$349,000**	**$0**	**$16,486,250**
1.4	**MONITORING AND INSTRUMENTATION**														
	Groundwater monitoring/seepage collection wells - NW Embankment	ea	$20,000	$40,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$40,000
	Groundwater monitoring/seepage collection wells - SE Embankment	ea	$20,000	$40,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$40,000
	Inclinometers - NW Embankment	ea	$15,000	$30,000	$0	$0	$0	$0	$15,000	$0	$0	$0	$15,000	$0	$60,000
	Inclinometers - SE Embankment	ea	$15,000	$30,000	$0	$0	$0	$0	$15,000	$0	$0	$0	$15,000	$0	$60,000
	Vibrating wire piezometers - NW Embankment	ea	$5,000	$175,000	$0	$0	$50,000	$0	$50,000	$0	$50,000	$0	$50,000	$0	$375,000
	Vibrating wire piezometers - SE Embankment	ea	$5,000	$175,000	$0	$0	$50,000	$0	$50,000	$0	$50,000	$0	$50,000	$0	$375,000
			Instrumentation Sub-Total	**$490,000**	**$0**	**$0**	**$100,000**	**$0**	**$130,000**	**$0**	**$100,000**	**$0**	**$130,000**	**$0**	**$950,000**
1.5	**WMF ENVIRONMENTAL COMPENSATION**														
	Fisheries Compensation	LS	$10,000,000	$10,000,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$10,000,000
			Compensation Sub-Total	**$10,000,000**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$10,000,000**
1.6	**CLOSURE AND RECLAMATION BOND**														
	5 Year Bond	LS	$8,500,000	$8,500,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$8,500,000
			and Reclamation Sub-Total	**$8,500,000**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$8,500,000**
	SUBTOTAL			$46,363,840	$0	$955,010	$1,295,120	$1,219,320	$1,537,110	$1,134,740	$1,609,670	$822,560	$560,090	$0	$55,497,460
	EPCM - EARTHWORKS, MECHANICAL, MONITORING AND INSTRUMENTATION AND WMF E		10%	$3,786,384	$0	$95,501	$129,512	$121,932	$153,711	$113,474	$160,967	$82,256	$56,009	$0	$4,699,746
	EPCM - CLOSURE AND RECLAMATION BOND		5%	$425,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$425,000
	MOBILIZATION/DEMOBILIZATION		8%	$3,709,107	$0	$76,401	$103,610	$97,546	$122,969	$90,779	$128,774	$65,805	$44,807	$0	$4,439,797
	INDIRECTS		20%	$9,272,768	$0	$191,002	$259,024	$243,864	$307,422	$226,948	$321,934	$164,512	$112,018	$0	$11,099,492
	CONTINGENCY - EARTHWORKS		25%	$2,608,035	$0	$238,753	$211,343	$304,830	$264,340	$283,685	$290,168	$205,640	$20,273	$0	$4,427,065
	CONTINGENCY - ELECTRICAL, MECHANICAL AND INSTRUMENTATION		20%	$3,486,340	$0	$0	$89,950	$0	$95,950	$0	$89,800	$0	$95,800	$0	$3,857,840
	CONTINGENCY - WMF ENVIRONMENTAL COMPENSATION		20%	$2,000,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$2,000,000
	CONTINGENCY - CLOSURE AND RECLAMATION BOND		20%	$1,700,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$1,700,000
	OVERALL PROJECT TOTAL			**$73,351,474**	**$0**	**$1,556,666**	**$2,088,558**	**$1,987,492**	**$2,481,502**	**$1,849,626**	**$2,601,312**	**$1,340,773**	**$888,997**	**$0**	**$88,146,400**

TMF and Infrastructure Costs - Adjusted

		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Spanish Mountain Gold														
TMF Cost Adjustment	H2 Tailings Option													
		Total	**Year -2**	**Year -1**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**
TMF Operating Cost														
Processed Ore	tonnes	116,905,974			10,800,000	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	12,124,406	10,105,983	9,193,317	2,682,268
Tailings Operating Cost	$	$ 13,410,000			$ 1,341,000	$ 1,341,000	$ 1,341,000	$ 1,341,000	$ 1,341,000	$ 1,341,000	$ 1,341,000	$ 1,341,000	$ 1,341,000	$ 1,341,000
Tailings Operating Unit Cost	$/tonne	$ 0.11			$ 0.12	$ 0.09	$ 0.09	$ 0.09	$ 0.09	$ 0.09	$ 0.11	$ 0.13	$ 0.15	$ 0.50
TMF Capital Cost		**Total**	**Year -2**	**Year -1**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**
Site Prep		$ 2,553,750	$ 1,935,000		$ -	$ 121,000	$ 90,750	$ 123,750	$ 82,500	$ 79,750	$ 46,750	$ 33,000	$ 41,250	$ -
Waste Management Facility		$ 14,579,510	$ 3,961,070	$ 3,961,070	$ -	$ 834,010	$ 754,620	$ 1,095,570	$ 974,860	$ 1,054,990	$ 1,113,920	$ 789,560	$ 39,840	$ -
Diversion Ditches		$ 575,000	$ 287,500	$ 287,500	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Electrical		$ 1,852,950		$ 1,852,950	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Mechanical		$ 16,486,250	$ 3,772,188	$ 11,316,563	$ -	$ -	$ 349,750	$ -	$ 349,750	$ -	$ 349,000	$ -	$ 349,000	$ -
Monitoring and Instrumentation		$ 950,000	$ 245,000	$ 245,000	$ -	$ -	$ 100,000	$ -	$ 130,000	$ -	$ 100,000	$ -	$ 130,000	$ -
Total		$ 36,997,460	$ 10,200,758	$ 17,663,083	$ -	$ 955,010	$ 1,295,120	$ 1,219,320	$ 1,537,110	$ 1,134,740	$ 1,609,670	$ 822,560	$ 560,090	$ -
Indirects														
EPCM	10.0%	$ 3,699,746	$ 1,020,076	$ 1,766,308	$ -	$ 95,501	$ 129,512	$ 121,932	$ 153,711	$ 113,474	$ 160,967	$ 82,256	$ 56,009	$ -
Mobilization/Demobilization	8.0%	$ 2,959,797	$ 816,061	$ 1,413,047	$ -	$ 76,401	$ 103,610	$ 97,546	$ 122,969	$ 90,779	$ 128,774	$ 65,805	$ 44,807	$ -
Indirects	20.0%	$ 7,399,492	$ 2,040,152	$ 3,532,617	$ -	$ 191,002	$ 259,024	$ 243,864	$ 307,422	$ 226,948	$ 321,934	$ 164,512	$ 112,018	$ -
Total Indirects	38.0%	$ 14,059,035	$ 3,876,288	$ 6,711,971	$ -	$ 362,904	$ 492,146	$ 463,342	$ 584,102	$ 431,201	$ 611,675	$ 312,573	$ 212,834	$ -
Contingency														
Earthworks	25.0%	$ 4,427,065	$ 1,545,893	$ 1,062,143	$ -	$ 238,753	$ 211,343	$ 304,830	$ 264,340	$ 283,685	$ 290,168	$ 205,640	$ 20,273	$ -
Electrical, Mechanical and Instrumentation	20.0%	$ 3,857,840	$ 803,438	$ 2,682,903	$ -	$ -	$ 89,950	$ -	$ 95,950	$ -	$ 89,800	$ -	$ 95,800	$ -
Total Contingency	22.4%	$ 8,284,905	$ 2,349,330	$ 3,745,045	$ -	$ 238,753	$ 301,293	$ 304,830	$ 360,290	$ 283,685	$ 379,968	$ 205,640	$ 116,073	$ -
Total TMF														
Direct		$ 36,997,460	$ 10,200,758	$ 17,663,083	$ -	$ 955,010	$ 1,295,120	$ 1,219,320	$ 1,537,110	$ 1,134,740	$ 1,609,670	$ 822,560	$ 560,090	$ -
Indirects		$ 14,059,035	$ 3,876,288	$ 6,711,971	$ -	$ 362,904	$ 492,146	$ 463,342	$ 584,102	$ 431,201	$ 611,675	$ 312,573	$ 212,834	$ -
Contingency		$ 8,284,905	$ 2,349,330	$ 3,745,045	$ -	$ 238,753	$ 301,293	$ 304,830	$ 360,290	$ 283,685	$ 379,968	$ 205,640	$ 116,073	$ -
		$ 59,341,400	$ 16,426,375	$ 28,120,099	$ -	$ 1,556,666	$ 2,088,558	$ 1,987,492	$ 2,481,502	$ 1,849,626	$ 2,601,312	$ 1,340,773	$ 888,997	$ -
PES Infrastructure Calculation		**Total**	**Year -2**	**Year -1**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**
Direct Infrastructure		$ 50,450,000	$ 37,127,500	$ 12,087,500	$ 1,235,000	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Indirects	20.0%	$ 10,090,000	$ 7,425,500	$ 2,417,500	$ 247,000	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Contingency	15.0%	$ 7,567,500	$ 5,569,125	$ 1,813,125	$ 185,250	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total		$ 68,107,500	$ 50,122,125	$ 16,318,125	$ 1,667,250	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Infrastructure and TMF Capital	% Calculated													
Direct		$ 87,447,460	$ 47,328,258	$ 29,750,583	$ 1,235,000	$ 955,010	$ 1,295,120	$ 1,219,320	$ 1,537,110	$ 1,134,740	$ 1,609,670	$ 822,560	$ 560,090	$ -
Indirects	27.62%	$ 24,149,035	$ 11,301,788	$ 9,129,471	$ 247,000	$ 362,904	$ 492,146	$ 463,342	$ 584,102	$ 431,201	$ 611,675	$ 312,573	$ 212,834	$ -
Contingency	18.13%	$ 15,852,405	$ 7,918,455	$ 5,558,170	$ 185,250	$ 238,753	$ 301,293	$ 304,830	$ 360,290	$ 283,685	$ 379,968	$ 205,640	$ 116,073	$ -
Total		$ 127,448,900	$ 66,548,500	$ 44,438,224	$ 1,667,250	$ 1,556,666	$ 2,088,558	$ 1,987,492	$ 2,481,502	$ 1,849,626	$ 2,601,312	$ 1,340,773	$ 888,997	$ -
Factor for total capital														
Indirect			0.47	0.38	0.01	0.02	0.02	0.02	0.02	0.02	0.03	0.01	0.01	-
Contingency			0.50	0.35	0.01	0.02	0.02	0.02	0.02	0.02	0.02	0.01	0.01	-

Operating Cost Calculations

Equipment and ManPower Calculations

Calendar Days	365
Scheduled Shutdown	
Unscheduled Days Down - weather	
Total	365
Shifts / Day	2
Scheduled Hours / Shift	12
Lunch Break	0.5
Shift Start / Shutdown	0.5
Coffee Breaks	0.5
Miscellaneous - Blasting & Moves	0.1
Standard Work Week	84
Weeks per year	26
number of shifts	4
Based Pay hours	2,184
Scheduled OT	1.00%
UnScheduled OT %	
OT payrate	1.5
Total Payhours	2,217
Vacation Allowance (hours)	120
Absenteeism %	3.00%
Sick time %	0.30%
Worked hours	2014

Pre-Production	months
Year -2	6
Year -1	12

Equipment Cycle times

min on dump	**1.00**
min at crusher	**1.00**

Duties and Import Taxes

Explosives Accessories	Waste		Ore		Wall Control	
	Unit Cost		Unit Cost		Unit Cost	
Boosters	$4.80	per booste	$4.80	per booste	$4.80	per booster
Downline		per metre		per metre		per metre
Trunkline	$0.69	per metre	$0.69	per metre	$0.69	per metre
Surface Delays	$5.70	per delay	$5.70	per delay	$5.70	per delay
DownHole Delays	$6.95	per delay	$6.95	per delay	$6.95	per delay
Initiation	$83.83	per blast	$83.83	per blast	$83.83	per blast
Miscellaneous including Liners	$5.00	per hole	$5.00	per hole	$5.00	per hole

Secondary Blasting & Development			
Explosives		of Primary Blast	
Accessories		of Primary Blast	

AN/FO to Emulsion Proportion (by volume)			
		Primary	Perimeter
AN/FO		0.75	0.75
Emulsion		0.25	0.25
Swell Factor			150%
Moisture			3%

General Blasting Related Costs (contract)			
Fixed Installations	number		
Explosives Magazine	1		/month
Accessories Magazine	1		/month
Pickup Trucks & Pumps & Labour	1	$54,600	/month

Explosives Type	
Ore (bulk/package)	bulk
Waste (bulk/package)	bulk

Explosives Package Costs	$/hole
Ore	
Waste	

AN/FO Fuel Calculations		
Ammonium Nitrate	$55.50	/100 kg
Fuel Price	$0.730	/litre
Fuel Density	0.84	kg/l
Fuel Price	$86.90	100 kg
Fuel Content	6.00%	by weight
Blended Price	$57.38	/100 kg
Emulsion costs	$63.50	/100 kg
Emulsion S.G.	1.2	

Est Mill Operating cost ($/t)		
Est G & A Cost (x 1000 $/year)		year
Mill Capital Estimate (year -1 $k)		-2
Other Capital Est (year -1 $k)		-1

Input data for Fuel/Power

Diesel	0.730	$/litre
Gasoline		$/litre
Electricity	0.045	$/kWh

Property Drilling Parameters

Drilling	Bit Diameter (mm)	
Ore Drilling	229.00	
Waste Drilling	229.00	1
Bench height - Ore	**10**	
Bench height - Waste	10	
Perimeter Drilling (Yes/Nc	**no**	
Secondary Drilling (% of		

	Loading Equipment	Capital Cost	Machine Life (in hrs)	# of units	Initial year required	Fuel Type (d/g/e)	Consumption (l/hr or	Lube, Oil & Filters (% of	Number of tires	Size	Life	Tire Price	Under Carriage	R&M Reserve	Special Wear Items	Operator Eff	dipper size	Bucket Fill Factor	Average Cycle time	Truck Spot Time	Static tipping	Trk Av. To Loader
1	L-1350	$4,350,000	35,000		-2	d	170.0	10%	4	55/80R57 & chains	5000	$96,250		$135.00	$75.00	83.3%	21.4	90%	38	42	75	80%
2	O&K 120C	$6,100,000	60,000		-1	d	180.0	10%						$381.32	$73.00	83.3%	17	95%	30	42	75	80%
3	O&K 170C	$7,900,000	62,000		-1	e	1100.0	10%						$278.20	$70.00	83.3%	22	95%	30	42	75	80%
4	O&K 200C	$12,100,000	60,000		-1	e	1100.0	10%						$334.55	$80.00	83.3%	26	95%	30	42	75	80%

	Trucking	Capital Cost	Machine Life (in hrs)	# of units	Initial year required	Fuel Type (d/g/e)	Consumption (l/hr or	Lube, Oil & Filters (% of	Number of tires	Size	Life	Tire Price	Under Carriage	R&M Reserve	Special Wear Items	Operator Eff	Size Capacity m3	Size Capacity (wmt)	Nominal Truck
1	Cat 793	$3,750,000	60,000		-2	d	170.0	10%	6	40.00 R57	5000	$33,500		$166.34	$9.75	83%	129 m³	218 t	240 t
2	Cat 789	$3,200,000	60,000		-2	d	140.0	10%	6	37.00 R57	5000	$27,600		$132.89	$6.44	83%	105 m³	185 t	200 t
3	Cat 785	$2,750,000	60,000		-2	d	101.0	10%	6	33.00 R51	5500	$19,000		$114.47	$4.78	83%	78 m³	144 t	150 t
4	Cat 777	$1,800,000	60,000		-2	d	74.0	10%	6	27.00 R49	6000	$11,800		$91.53	$3.68	83%	60 m³	91 t	100 t
5																83%	60 m³	60 t	125 t

	Drilling Equipment	Capital Cost	Machine Life (in hrs)	# of units	Initial year required	Fuel Type (d/g/e)	Consumption (l/hr or KW/hr)	Lube, Oil & Filters (% of fuel)	Drill Bits Unit Costs	Drill Bit Life (metres)			Under Carriage	R&M Reserve	Special Wear Items	Operator Eff	Scheduled Downtime (in shifts)	UnScheduled Downtime (in %)	Drill Type	Bit Diameter (mm)	Down Pressure	Move, Spot and Collar hole	Level Drill (in min)	Add Steel (in min)	Pull Rods (in min)	Penetration Rate Ore (metres/min)	Penetration Rate Waste (metres/min)	RPM
1	Primary Drill	$1,600,000	26,000		-1	d	170.0	10%	$5,154	1,100			$6.00	$66.00		83.3%	52		Rotary	229.00	100,000	3.00	0.25		0.50	0.49	0.49	70
2	Primary Drill	$1,600,000	26,000		2	d	170.0	10%	$5,154	1,100			$6.00	$66.00		83.3%	52		Rotary	229.00	100,000	3.00	0.25		0.50	0.49	0.49	70
3																83.3%											0.60	

	Support	Capital Cost	Machine Life (in hrs)	# of units	Initial year required	Fuel Type (d/g/e)	Consumption (l/hr or KW/hr)	Lube, Oil & Filters (% of fuel)	Number of tires	Size	Life	Tire Price	Under Carriage	R&M Reserve	Special Wear Items	Operator Eff	Scheduled Downtime	UnScheduled Downtime
1	Track Dozer	$1,500,000	35,000	2	-2	d	75.0	10%					$20.00	$76.95	$5.00	83.3%	26	11.4%
2	Grader	$800,000	20,000	2	-2	d	33.0	10%	6	18.00-25 12 PR	4000	$3,300		$51.37	$11.00	83.3%	26	11.4%
3	Rubber Tired Dozer	$1,300,000	30,000	1	-1	d	55.0	10%	4	29.5R25	5250	$11,100		$46.28	$2.25	83.3%	26	11.4%
4	Transfer Loader	$1,000,000	20,000	1	-1	d	48.0	10%	4	35/65R33	5250	$14,500		$55.46	$9.00	83.3%	26	11.4%
5	Backhoe with hammer	$ 500,000	10	1	-1	d	31.0	10%					$7.00	$30.36		83.3%	26	11.4%
6	Water Truck	$ 260,000	10	2	-1	d	20.0	10%	6	10.00-20PR16	2000	$500		$10.00		83.3%	26	11.4%
7					-1											83.3%	26	11.4%
8					-1													

	Mine General Equipment													
1	Lube/Fuel Truck	$160,000	6	1	-2	d	20.0	0.1	6	11R22.5	2000	$700		$5.00
2	Mechanic's Truck	$216,000	4		-2	d	20.0	0.1	6	11R22.5	2000	$700		$5.00
3	Welding Truck	$208,000	6		-2	d	20.0	0.1	6	11R22.5	2000	$700		$5.00
4	Blasting Loader	$65,000	5	2	-2	d	10.0	0.1	6	12 x 16.5	2000	$250		$10.00
5	Blasters Truck	$50,000	5	2	-2	d	10.0	0.1	4	265/75 R16	2000	$300		$5.00
6	Integrated Tool Carrier	$256,000	10	1	-2	d	18.0	0.3	4	20.5R25	3000	$2,900		$10.00
7	Compactor	$170,000	10	1	-2	d	10.0	0.1						$10.00
8	Lighting Plants	$17,000	4	7	-2	d	6.0	0.1	2	265/75 R16	4000	$300		$2.00
9	ANFO Truck				-1	d		0.1						
10	Auxilary Pumps	$45,000	5	2	-1	d	10.0							$10.00
11	Pump Lines													
12	Man Bus	$80,000	5	3	-2	d	9.0	0.1	6		2000	$300		$5.00
13	Pickup Trucks	$50,000	2	6	-2	d	5.0	0.3	4	265/75 R16	2000	$300		$5.00
2					-1									

	Fuel $/hr	Power $/Ophr	Lube, Oil & Filters	Tires $/hr	Under-Carriage	R&M Reserve	Special Wear Items	Total	Drill Bits m/hr	Drill Bits $/hr	Drill Bits $/m
Drills											
Primary Drill	$124.10		$12.41		$6.00	$66.00	$118.54	**$327.05**	25.3	$118.54	$4.69
Primary Drill	$124.10		$12.41		$6.00	$66.00		**$208.51**			$4.69
Loading Equipment									Tire unit cost	$/hr	
L-1350	$124.10		$12.41	$77.00		$135.00	$75.00	**$423.51**	$96,250	$77.00	
O&K 120C	$131.40		$13.14			$381.32	$73.00	**$598.86**			
O&K 170C		$49.50				$278.20	$70.00	**$397.70**			
O&K 200C		$49.50				$334.55	$80.00	**$464.05**			
Hauling Equipment											
Cat 793	$124.10		$12.42	$40.20		$166.34	$9.75	**$352.81**	$33,500	$40.20	
Cat 789	$102.20		$10.22	$33.12		$132.89	$6.44	**$284.87**	$27,600	$33.12	
Cat 785	$73.73		$7.37	$20.73		$114.47	$4.78	**$221.08**	$19,000	$20.73	
Cat 777	$54.02		$5.40	$11.80		$91.53	$3.68	**$166.43**	$11,800	$11.80	
Mine Support Equipment											
Track Dozer	$54.75		$5.48		$20.00	$76.95	$5.00	**$162.18**			
Grader	$24.09		$2.41	$4.95		$51.37	$11.00	**$93.82**	$3,300	$4.95	
Rubber Tired Dozer	$40.15		$4.02	$8.46		$46.28	$2.25	**$101.15**	$11,100	$8.46	
Backhoe with hammer	$22.63		$2.26		$7.00	$30.36		**$62.25**			
Water Truck	$14.60		$1.46	$1.50		$10.00		**$27.56**	$500	$1.50	
Mine General Equipment											
Lube/Fuel Truck	$14.60		$1.46	$2.10		$5.00		**$23.16**	$700	$2.10	
Mechanic's Truck	$14.60		$1.46	$2.10		$5.00		**$23.16**	$700	$2.10	
Welding Truck	$14.60		$1.46	$2.10		$5.00		**$23.16**	$700	$2.10	
Blasting Loader	$7.30		$0.73	$0.75		$10.00		**$18.78**	$250	$0.75	
Blasters Truck	$7.30		$0.73	$0.60		$5.00		**$13.63**	$300	$0.60	
Integrated Tool Carrier	$13.14		$3.29	$3.87		$10.00		**$30.29**	$2,900	$3.87	
Compactor	$7.30		$0.73			$10.00		**$18.03**			
Lighting Plants	$4.38		$0.44	$0.15		$2.00		**$6.97**	$300	$0.15	
Auxilary Pumps	$7.30					$10.00		**$17.30**			
Man Bus	$6.57		$0.66	$0.90		$5.00		**$13.13**	$300	$0.90	
Pickup Trucks	3.65		$0.91	$0.60		$5.00		**$10.16**	$300	$0.60	

	-2	-1	1	2	3	4	5	6	7	8	9	10
Mine Staff Manpower Requirments												
INE MAINTENANCE												
Maintenance Superintendent	1	1	1	1	1	1	1	1	1	1		
Maintenance General Foreman												
Maintenance Shift Foremen												
Maintenance Planner	1	2	2	2	2	2	2	2	2	2	2	1
Clerk/Secretary	1	1	1	1	1	1	1	1	1	1	1	1
Subtotal	**3**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**
INE OPERATIONS												
Mine Operations Superintendent	1	1	1	1	1	1	1	1	1	1	1	1
Mine General Foreman	1	1	1	1	1	1	1	1	1	1	1	
Mine Shift Foreman	4	4	4	4	4	4	4	4	4	4	4	4
Drill and Blast Foreman	1	1	1	1	1	1	1	1	1	1	1	
Training Foreman	1	1	1	1	1	1	1	1	1	1		
Clerk/Secretary	1	1	1	1	1	1	1	1	1	1	1	1
Subtotal	**9**	**9**	**9**	**9**	**9**	**9**	**9**	**9**	**9**	**9**	**9**	**9**
INE ENGINEERING												
Chief Engineer	1	1	1	1	1	1	1	1	1	1	1	
Senior Engineer	1	1	1	1	1	1	1	1	1	1	1	1
Open Pit Planning Engineer	1	1	1	1	1	1	1	1	1	1	1	
Surveyor/Mining Technician	3	3	3	3	3	3	3	3	3	3	3	2
Clerk/Secretary	1	1	1	1	1	1	1	1	1	1	1	1
Subtotal	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**	**7**
EOLOGY												
Chief Geologist	1	1	1	1	1	1	1	1	1	1		
Senior Geologist	1	1	1	1	1	1	1	1	1	1	1	
Grade Control Geologist	1	1	1	1	1	1	1	1	1	1	1	1
Sampling Technician	2	2	2	2	2	2	2	2	2	2	2	1
Clerk/Secretary	1	1	1	1	1	1	1	1	1	1	1	1
Subtotal	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**5**	**3**
OTAL MINE STAFF	**25**	**26**	**26**	**26**	**26**	**26**	**26**	**26**	**26**	**26**	**25**	**23**

	Mine Manpower Requirements											
	-2	-1	1	2	3	4	5	6	7	8	9	10
MINE GENERAL												
Operations												
Tool Crib Attendent	1	1	1	1	1	1	1	1	1	1	1	
Warehouse Attendent	2	2	2	2	2	2	2	2	2	2	2	1
General Mine Labourer	2	2	2	2	2	2	2	2	2	2	2	2
Trainee	2	2	2	2	2	2	2	2	2	2		
Maintenance												
Light Duty Mechanic	2	2	2	2	2	2	2	2	2	2	1	1
Tire Man	4	4	4	4	4	4	4	4	4	4	2	1
Lube Truck Driver	4	4	4	4	4	4	4	4	4	4	4	2
Subtotal	**17**	**17**	**17**	**17**	**17**	**17**	**17**	**17**	**17**	**17**	**12**	**7**
DRILLING												
Operations												
Drill Operator 1	0.90	4	4	4	4	4	4	4	4	8	4	4
Drill Operator 2				4	4	4	4	4	8	8	4	4
Maintenance												
Heavy Duty Mechanic	0.2											
Welder												
Electrician												
Subtotal	**1**	**4**	**4**	**8**	**8**	**8**	**8**	**8**	**12**	**16**	**8**	**8**
BLASTING												
Operations												
Blasters	2	2	2	2	2	2	2	2	2	2	1	1
Blaster Helper	2	2	2	2	2	2	2	2	2	2	2	2
Subtotal	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**3**	**3**
LOADING												
Operations												
L-1350	4	4	4	4	4	4	4	4	4	4	8	4
O&K 120C												
O&K 170C		4	4	8	8	8	8	8	8	8	4	4
O&K 200C												
Maintenance												
Heavy Duty Mechanic												
Welder												
Electrician												
Subtotal		**8**	**8**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**12**	**8**
HAULING												
Operations												
Haulage Truck Driver												
Haulage Truck Driver	18	14	17	37	47	52	51	45	35	30	26	12
Haulage Truck Driver												
Haulage Truck Driver												
Maintenance												
Heavy Duty Mechanic												
Welder												
Subtotal		**14**	**17**	**37**	**47**	**52**	**51**	**45**	**35**	**30**	**26**	**12**
MINE OPERATIONS SUPPORT												
Operations												
Dozer Operator	5.0	4.0	5.0	7.0	7.0	7.0	7.0	7.0	7.0	7.0	7.0	2.0
Grader / RT Operator	1.9	2.0	3.6	6.3	7.3	7.8	7.7	7.1	6.0	5.5	4.4	0.8
Water Truck Driver	0.4	0.7	0.8	1.8	2.2	2.2	2.2	2.2	1.7	1.5	1.3	0.4
Backhoe Operator	1.9	3.1	5.0	6.0	6.0	6.0	6.0	5.0	5.0	5.0	4.0	2.0
Maintenance												
Heavy Duty Mechanic												
Welder												
Apprentice												
Subtotal	**9.2**	**9.8**	**14.4**	**21.1**	**22.5**	**23**	**22.9**	**21.3**	**19.7**	**19**	**16.7**	**5.2**
MINE SUMMARY												
Operations Subtotal	41	45	52	87	99	104	103	95	88	86	71	39
Maintenance Subtotal	10	10	10	10	10	10	10	10	10	10	7	2
Total	**51**	**55**	**62**	**97**	**109**	**114**	**113**	**105**	**98**	**96**	**78**	**41**

		-2	-1	1	2	3	4	5	6	7	8	9	10	PRE-PRODUCTION	PRODUCTION	TOTAL
													6.0			
MINE OPERATIONS - PRODUCTION															**From Yr 1**	
	days	365	365	365	365	365	365	365	365	365	365	365	91.25	**730**	**3,376**	**4,106**
ORE Delivered	(kt)	2,820	7,422	13,043	15,622	15,625	15,622	15,635	15,622	23,282	30,861	22,109	4,708	**10,242**	**172,128**	**182,370**
Ore Milled	(kt)			10,800	14,400	14,400	14,400	14,400	14,400	12,124	10,106	9,193	2,682		**116,906**	**116,906**
West	(kt)	997	2,182	11,028	29,099	30,693	28,556	29,325	18,550	8,426				**3,179**	**155,677**	**158,857**
NW Tail	(kt)	1,918	1,918		575	826	1,289	1,123	1,283	1,420	1,031			**3,835**	**7,547**	**11,382**
TOTAL WASTE	(kt)	2,915	4,100	11,028	29,674	31,520	29,846	30,448	19,833	9,845	1,031			**7,015**	**163,224**	**170,239**
TOTAL	(kt)	5,735	11,522	24,071	45,296	47,145	45,468	46,082	35,455	33,127	31,892	22,109	4,708	**17,257**	**335,352**	**352,609**
TOTAL DAILY PRODUCTION	(kt/day)	15.7	31.6	65.9	124.1	129.2	124.6	126.3	97.1	90.8	87.4	60.6	51.6	**23.6**	**99.3**	**85.9**
STRIP RATIO	w/o	1.03	0.55	0.85	1.90	2.02	1.91	1.95	1.27	0.42	0.03			**0.68**	**0.95**	**0.93**
MINE OPERATING COST FORECAST																
GENERAL MINE & ENGINEERING																
Salaries & Wages	Staff	1,737.1	2,538.7	2,538.7	2,538.7	2,538.7	2,538.7	2,538.7	2,538.7	2,538.7	2,538.7	2,185.9	339.8	4,275.8	22,834.9	27,110.7
Salaries & Wages	Labour	511.2	1,314.9	1,314.9	1,314.9	1,314.9	1,314.9	1,314.9	1,314.9	1,314.9	1,314.9	918.2	132.0	1,826.1	11,569.3	13,395.4
Fuel & Power	($ x 1000)															
Dewatering	($ x 1000)			200.0	200.0	200.0	200.0	200.0	200.0	200.0	100.0	100.0	100.0		1,700.0	1,700.0
Consumables, R&M Parts	($ x 1000)	115.0	115.0	115.0	115.0	115.0	115.0	115.0	115.0	115.0	115.0	115.0	115.0	230.0	1,150.0	1,380.0
Subtotal	($ x 1000)	**2,363.3**	**3,968.5**	**4,168.5**	**4,168.5**	**4,168.5**	**4,168.5**	**4,168.5**	**4,168.5**	**4,168.5**	**4,068.5**	**3,319.1**	**686.8**	**6,331.9**	**37,254.2**	**43,586.1**
DRILLING																
Salaries & Wages	($ x 1000)	48.1	338.5	338.5	677.1	677.1	677.1	677.1	677.1	1,015.6	1,354.2	677.1	169.3	386.6	6,940.1	7,326.7
Fuel & Power	($ x 1000)	199.4	398.1	397.5	646.3	708.5	737.0	730.2	736.8	1,084.5	1,393.1	968.0	206.9	597.5	7,608.8	8,206.4
Consumables, R&M Parts	($ x 1000)	294.3	587.8	586.9	957.7	1,050.1	1,092.1	1,082.3	1,091.6	1,610.2	2,064.8	1,434.7	306.7	882.1	11,277.0	12,159.1
Subtotal	($ x 1000)	**541.8**	**1,324.5**	**1,323.0**	**2,281.1**	**2,435.6**	**2,506.2**	**2,489.5**	**2,505.5**	**3,710.3**	**4,812.1**	**3,079.8**	**682.8**	**1,866.3**	**25,825.9**	**27,692.2**
BLASTING																
Salaries & Wages	($ x 1000)	157.8	315.5	315.5	315.5	315.5	315.5	315.5	315.5	315.5	315.5	230.9	57.7	473.3	2,812.8	3,286.1
Consumables & Direct Costs	($ x 1000)	990.2	1,992.6	2,004.3	2,807.5	3,014.4	3,104.7	3,084.8	3,103.9	4,234.3	5,296.2	3,884.5	851.5	2,982.8	31,386.1	34,368.9
Subtotal	($ x 1000)	**1,147.9**	**2,308.1**	**2,319.8**	**3,123.0**	**3,330.0**	**3,420.3**	**3,400.3**	**3,419.4**	**4,549.9**	**5,611.7**	**4,115.4**	**909.2**	**3,456.0**	**34,198.9**	**37,654.9**
LOADING																
Salaries & Wages	($ x 1000)	172.7	690.9	690.9	1,036.4	1,036.4	1,036.4	1,036.4	1,036.4	1,036.4	1,036.4	1,036.4	172.7	863.7	9,154.9	10,018.5
Fuel & Power	($ x 1000)	323.5	566.7	661.2	1,150.9	1,226.9	1,184.5	1,200.0	823.1	757.6	747.8	646.8	137.8	890.2	8,536.5	9,426.7
Consumables, R&M Parts	($ x 1000)	780.4	1,541.4	3,074.1	5,732.6	5,969.5	5,758.0	5,835.2	4,458.2	4,162.3	4,012.9	2,821.4	600.9	2,321.8	42,425.0	44,746.8
Subtotal	($ x 1000)	**1,276.6**	**2,799.0**	**4,426.2**	**7,919.9**	**8,232.9**	**7,978.9**	**8,071.7**	**6,317.6**	**5,956.3**	**5,797.0**	**4,504.5**	**911.4**	**4,075.6**	**60,116.4**	**64,192.1**
HAULING																
Salaries & Wages	($ x 1000)	689.3	1,072.2	1,302.0	2,833.8	3,599.7	3,982.6	3,906.0	3,446.5	2,680.6	2,297.7	1,991.3	229.8	1,761.5	26,270.0	28,031.6
Fuel & Power	($ x 1000)	1,296.3	2,306.0	2,835.7	6,207.7	7,963.9	8,779.5	8,537.1	7,536.8	5,877.5	5,088.1	4,353.3	1,188.0	3,602.3	58,367.6	61,970.0
Consumables, R&M Parts	($ x 1000)	2,316.9	4,121.8	5,068.4	11,095.5	14,234.4	15,692.3	15,259.0	13,471.2	10,505.4	9,094.4	7,781.0	2,123.4	6,438.7	104,325.0	110,763.7
Subtotal	($ x 1000)	**4,302.5**	**7,500.1**	**9,206.1**	**20,137.1**	**25,798.0**	**28,454.4**	**27,702.2**	**24,454.6**	**19,063.5**	**16,480.2**	**14,125.6**	**3,541.1**	**11,802.6**	**188,962.7**	**200,765.3**
SUPPORT																
Salaries & Wages	($ x 1000)	381.4	812.5	1,193.8	1,749.3	1,865.3	1,906.8	1,898.5	1,765.8	1,633.2	1,575.2	1,384.5	107.8	1,193.8	15,080.1	16,273.9
Fuel & Power	($ x 1000)	1,851.1	947.0	1,133.0	1,496.6	1,546.2	1,565.4	1,559.7	1,516.8	1,457.2	1,424.4	1,237.6	371.3	2,798.2	13,308.1	16,106.3
Consumables, R&M Parts	($ x 1000)	3,423.5	1,777.9	1,929.9	2,661.1	2,788.2	2,843.9	2,827.3	2,725.2	2,586.3	2,513.8	2,210.4	697.7	5,201.4	23,783.8	28,985.2
Subtotal	($ x 1000)	**5,656.0**	**3,537.4**	**4,256.7**	**5,906.9**	**6,199.7**	**6,316.0**	**6,285.5**	**6,007.8**	**5,676.6**	**5,513.3**	**4,832.5**	**1,176.8**	**9,193.4**	**52,172.0**	**61,365.3**
SUMMARY																
Salaries & Wages	($ x 1000)	3,697.6	7,083.2	7,694.4	10,465.6	11,347.6	11,771.9	11,687.1	11,094.9	10,534.9	10,432.5	8,424.3	1,209.0	10,780.8	94,662.1	105,442.9
Fuel & Power	($ x 1000)	3,670.3	4,217.9	5,027.3	9,501.5	11,445.5	12,266.5	12,027.0	10,613.5	9,176.8	8,653.4	7,205.6	1,904.0	7,888.2	87,821.1	95,709.3
Consumables	($ x 1000)	7,920.3	10,136.4	12,778.6	23,369.4	27,171.7	28,605.9	28,203.7	24,965.1	23,213.5	23,097.0	18,247.0	4,695.2	18,056.7	214,347.0	232,403.7
Subtotal	($ x 1000)	**15,288.2**	**21,437.5**	**25,500.3**	**43,336.5**	**49,964.7**	**52,644.3**	**51,917.8**	**46,673.4**	**42,925.2**	**42,182.9**	**33,876.9**	**7,808.2**	**36,725.7**	**396,830.2**	**433,555.9**
MINE - UNIT OPERATING COST																
General Mine Expense	$/t Mined	$0.4121	$0.3444	$0.1732	$0.0920	$0.0884	$0.0917	$0.0905	$0.1176	$0.1258	$0.1276	$0.1501	$0.1459	$0.37	$0.11	$0.124
Drilling	$/t Mined	$0.0945	$0.1149	$0.0550	$0.0504	$0.0517	$0.0551	$0.0540	$0.0707	$0.1120	$0.1509	$0.1393	$0.1450	$0.11	$0.08	$0.079
Blasting	$/t Mined	$0.2002	$0.2003	$0.0964	$0.0689	$0.0706	$0.0752	$0.0738	$0.0964	$0.1373	$0.1760	$0.1861	$0.1931	$0.20	$0.10	$0.107
Loading	$/t Mined	$0.2226	$0.2429	$0.1839	$0.1748	$0.1746	$0.1755	$0.1752	$0.1782	$0.1798	$0.1818	$0.2037	$0.1936	$0.24	$0.18	$0.182
Hauling	$/t Mined	$0.7503	$0.6509	$0.3825	$0.4446	$0.5472	$0.6258	$0.6011	$0.6897	$0.5755	$0.5168	$0.6389	$0.7521	$0.68	$0.56	$0.569
Support	$/t Mined	$0.9863	$0.3070	$0.1768	$0.1304	$0.1315	$0.1389	$0.1364	$0.1695	$0.1714	$0.1729	$0.2186	$0.2500	$0.53	$0.16	$0.174
Total Mine Operations	$/t Mined	**$2.6659**	**$1.8606**	**$1.0594**	**$0.9567**	**$1.0598**	**$1.1578**	**$1.1266**	**$1.3164**	**$1.2958**	**$1.3227**	**$1.5323**	**$1.6585**	**$2.13**	**$1.18**	**$1.23**

Spanish Mountain Gold

40K Mine Operating Cost Adjustment

		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Mined Tonnage														
Ore to Mill	tonnes	111,327,446	-	-	6,875,081	12,962,213	14,184,178	14,400,000	14,400,000	14,400,000	12,124,406	10,105,983	9,193,317	2,682,268
Ore to Stockpile	tonnes	5,578,528	1,646,642	3,931,886	-	-	-	-	-	-	-	-	-	-
Stockpile to Mill	tonnes	5,578,528	-	-	3,924,919	1,437,787	215,822	-	-	-	-	-	-	-
Waste	tonnes	230,126,161	4,088,038	7,590,346	13,272,973	30,896,338	32,744,747	31,067,578	31,682,172	21,054,746	21,002,742	21,785,538	12,915,230	2,025,713
Total	tonnes	352,610,663	5,734,680	11,522,232	24,072,973	45,296,338	47,144,747	45,467,578	46,082,172	35,454,746	33,127,148	31,891,521	22,108,547	4,707,981
Mine Operating Cost	$	$ 433,555,924	$ 15,288,205	$ 21,437,532	$ 25,500,250	$ 43,336,499	$ 49,964,696	$ 52,644,319	$ 51,917,782	$ 46,673,449	$ 42,925,207	$ 42,182,864	$ 33,876,918	$ 7,808,202
Sampling Cost	$	$ 4,370,760	$ 110,720	$ 221,520	$ 221,520	$ 343,200	$ 376,760	$ 392,240	$ 388,680	$ 392,080	$ 563,240	$ 739,680	$ 512,760	$ 108,360
Total Mine Operating Cost	**$**	**$ 437,926,684**	**$ 15,398,925**	**$ 21,659,052**	**$ 25,721,770**	**$ 43,679,699**	**$ 50,341,456**	**$ 53,036,559**	**$ 52,306,462**	**$ 47,065,529**	**$ 43,488,447**	**$ 42,922,544**	**$ 34,389,678**	**$ 7,916,562**
	$/t	1.26												
Sampling Cost Calculation														
Sample Cost	$/sample	$ 560	$ 40	$ 40	$ 40	$ 40	$ 40	$ 40	$ 40	$ 40	$ 40	$ 40	$ 40	$ 40
Number of Blastholes	# of samples	109,269	2768	5538	5538	8580	9419	9806	9717	9802	14081	18492	12819	2709
Sampling Cost	$	$ 4,370,760	$ 110,720	$ 221,520	$ 221,520	$ 343,200	$ 376,760	$ 392,240	$ 388,680	$ 392,080	$ 563,240	$ 739,680	$ 512,760	$ 108,360

Spanish Mountain Gold		Interest Rate		
Spanish Mountain		3%		
Open Pit Mining Capital	**Capital Unit Cost**	**Lease Period (Yr)**	**Lease Term Interest**	**Total Lease Cost**
Production Drill	$ 1,500,000	5	116%	$ 1,738,911
Loader - 11.5 m3	$ 1,600,000			$ -
Loader - 17 m3	$ 4,900,000			$ -
Loader - 21 m3 (L-1350)	$ 4,350,000	5	116%	$ 5,042,842
Hydraulic Shovel - 17 m3 O&K 120C	$ 4,800,000			$ -
Hydraulic Shovel - 21 m3 O&K 170C	$ 7,900,000	5	116%	$ 9,158,265
Hydraulic Shovel - 28 m3 O&K 200C	$ 10,000,000			$ -
Hydraulic Shovel - 34 m3 O&K 340C	$ 12,600,000			$ -
Breaker Loader - 6.5 cubic metre	$ 1,000,000	5	116%	$ 1,159,274
				$ -
Haulage Trucks (240 ton)	$ 3,750,000			$ -
Haulage Trucks (200 ton)	$ 3,200,000	5	116%	$ 3,709,677
Haulage Trucks (150 ton)	$ 2,750,000			$ -
Haulage Trucks (100 ton)	$ 1,800,000			$ -
				$ -
Tracked Dozer (433 kW)	$ 1,500,000			$ -
Tracked Dozer (306 kW)	$ 1,100,000	5	116%	$ 1,275,201
Tracked Dozer (231 kW)	$ 875,000			$ -
Grader (233 kW)	$ 800,000	5	116%	$ 927,419
Rubber Tired Dozer (350 kW)	$ 1,300,000	5	116%	$ 1,507,056
Utility Backhoe with hammer (2.3 cubic metre)	$ 505,000	5	116%	$ 585,433
Water Truck (Sterling)	$ 290,000	5	116%	$ 336,189
Tool Carrier	$ 350,000	5	116%	$ 405,746
Blasting Skid Steer Loader	$ 65,000	2	106%	$ 68,959
Light Plants	$ 17,000	2	106%	$ 18,035
Lube/Fuel Truck	$ 310,000	5	116%	$ 359,375
Mechanics Truck	$ 230,000	2	106%	$ 244,007
Welding Truck	$ 220,000	2	106%	$ 233,398
Crewcab Pickups	$ 52,000	2	106%	$ 55,167
Blasters Truck	$ 52,000	2	106%	$ 55,167
Pumps	$ 45,000	2	106%	$ 47,741
Pickup Truck	$ 46,000	2	106%	$ 48,801
Manbus	$ 80,000	2	106%	$ 84,872
Ambulance	$ 100,000	2	106%	$ 106,090
Fire Truck	$ 260,000	2	106%	$ 275,834
Compactor	$ 260,000	5	116%	$ 301,411
Lowboy	$ 100,000	2	106%	$ 106,090

	Unit Cost $	Total Capital Cost $Cdn	Units Required											
			Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Total Capital Cost														
Open Pit Mining Lease		$ 99,571,237												
Open Pit Mining Capital	**Lease Cost**	**Fleet Cost**												
Production Drill	$ 1,738,911	$ 6,955,644	1		1					2				
Loader - 11.5 m3	$ -	$ -												
Loader - 17 m3	$ -	$ -												
Loader - 21 m3 (L-1350)	$ 5,042,842	$ 5,042,842	1											
Hydraulic Shovel - 17 m3 O&K 120C	$ -	$ -												
Hydraulic Shovel - 21 m3 O&K 170C	$ 9,158,265	$ 18,316,530		1	1									
Hydraulic Shovel - 28 m3 O&K 200C	$ -	$ -												
Hydraulic Shovel - 34 m3 O&K 340C	$ -	$ -												
Breaker Loader - 6.5 cubic metre	$ 1,159,274	$ 2,318,548		1				1						
	$ -													
Haulage Trucks (240 ton)	$ -	$ -												
Haulage Trucks (200 ton)	$ 3,709,677	$ 48,225,801	4	1	5	2	1							
Haulage Trucks (150 ton)	$ -	$ -												
Haulage Trucks (100 ton)	$ -	$ -												
	$ -	$ -												
Tracked Dozer (433 kW)	$ -	$ -												
Tracked Dozer (306 kW)	$ 1,275,201	$ 6,376,007	2		1					2				
Tracked Dozer (231 kW)	$ -	$ -												
Grader (233 kW)	$ 927,419	$ 3,709,677	2							2				
Rubber Tired Dozer (350 kW)	$ 1,507,056	$ 3,014,113	1							1				
Utility Backhoe with hammer (2.3 cubic metre)	$ 585,433	$ 585,433	1											
Water Truck (Sterling)	$ 336,189	$ 672,379	2											
Tool Carrier	$ 405,746	$ 405,746	1											
Blasting Skid Steer Loader	$ 68,959	$ 275,834	2					2						
Light Plants	$ 18,035	$ 378,741	7				7				7			
Lube/Fuel Truck	$ 359,375	$ 359,375	1											
Mechanics Truck	$ 244,007	$ -												
Welding Truck	$ 233,398	$ -												
Crewcab Pickups	$ 55,167	$ 662,002	2		2		2		2		2			
Blasters Truck	$ 55,167	$ 165,500	1					1						
Pumps	$ 47,741	$ 477,405	2		2			2		2				
Pickup Truck	$ 48,801	$ 585,617	2		2		2		2		2			
Manbus	$ 84,872	$ 254,616	1					1						
Ambulance	$ 106,090	$ 106,090	1											
Fire Truck	$ 275,834	$ 275,834	1											
Compactor	$ 301,411	$ 301,411	1											
Lowboy	$ 106,090	$ 106,090	1											

	Unit Cost $	Total Capital Cost $Cdn	Units Under Lease											
			Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Total Capital Cost														
Open Pit Mining Lease		$ 99,571,237												
Open Pit Mining Capital	**Lease Cost**	**Fleet Cost**												
Production Drill	$ 1,738,911	$ 6,955,644	1	1	2	2	2	1	1	2	2	2	2	2
Loader - 11.5 m3	$ -	$ -												
Loader - 17 m3	$ -	$ -												
Loader - 21 m3 (L-1350)	$ 5,042,842	$ 5,042,842	1	1	1	1	1							
Hydraulic Shovel - 17 m3 O&K 120C	$ -	$ -												
Hydraulic Shovel - 21 m3 O&K 170C	$ 9,158,265	$ 18,316,530		1	2	2	2	2	1					
Hydraulic Shovel - 28 m3 O&K 200C	$ -	$ -												
Hydraulic Shovel - 34 m3 O&K 340C	$ -	$ -												
Breaker Loader - 6.5 cubic metre	$ 1,159,274	$ 2,318,548			1	1	1	1	1	1	1	1	1	1
	$ -													
Haulage Trucks (240 ton)	$ -	$ -												
Haulage Trucks (200 ton)	$ 3,709,677	$ 48,225,801	4	5	10	12	13	9	8	3	1			
Haulage Trucks (150 ton)	$ -	$ -												
Haulage Trucks (100 ton)	$ -	$ -												
	$ -	$ -												
Tracked Dozer (433 kW)	$ -	$ -												
Tracked Dozer (306 kW)	$ 1,275,201	$ 6,376,007	2	2	3	3	3	1	1	2	2	2	2	2
Tracked Dozer (231 kW)	$ -	$ -												
Grader (233 kW)	$ 927,419	$ 3,709,677	2	2	2	2	2			2	2	2	2	2
Rubber Tired Dozer (350 kW)	$ 1,507,056	$ 3,014,113	1	1	1	1	1			1	1	1	1	1
Utility Backhoe with hammer (2.3 cubic metre)	$ 585,433	$ 585,433	1	1	1	1	1							
Water Truck (Sterling)	$ 336,189	$ 672,379	2	2	2	2	2							
Tool Carrier	$ 405,746	$ 405,746	1	1	1	1	1							
Blasting Skid Steer Loader	$ 68,959	$ 275,834	2	2				2	2					
Light Plants	$ 18,035	$ 378,741	7	7			7	7			7	7		
Lube/Fuel Truck	$ 359,375	$ 359,375	1	1	1	1	1							
Mechanics Truck	$ 244,007	$ -												
Welding Truck	$ 233,398	$ -												
Crewcab Pickups	$ 55,167	$ 662,002	2	2	2	2	2	2	2	2	2	2	2	2
Blasters Truck	$ 55,167	$ 165,500	1	1				1	1				1	1
Pumps	$ 47,741	$ 477,405	2	2	2	2		2	2	2	2		2	2
Pickup Truck	$ 48,801	$ 585,617	2	2	2	2	2	2	2	2	2	2	2	2
Manbus	$ 84,872	$ 254,616	1	1				1	1				1	1
Ambulance	$ 106,090	$ 106,090	1	1										
Fire Truck	$ 275,834	$ 275,834	1	1										
Compactor	$ 301,411	$ 301,411	1	1	1	1	1							
Lowboy	$ 106,090	$ 106,090	1	1										

	Unit Cost $	Total Capital Cost $Cdn	Lease Cost											
			Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Total Capital Cost														
Open Pit Mining Lease		$ 99,571,237	$ 6,569,238	$ 9,142,826	$ 15,072,725	$ 16,556,596	$ 17,313,915	$ 11,529,212	$ 8,892,500	$ 4,487,394	$ 3,066,646	$ 2,276,971	$ 2,331,607	$ 2,331,607
Open Pit Mining Capital	**Lease Cost**	**Fleet Cost**												
Production Drill	$ 1,738,911	$ 6,955,644	$ 347,782	$ 347,782	$ 695,564	$ 695,564	$ 695,564	$ 347,782	$ 347,782	$ 695,564	$ 695,564	$ 695,564	$ 695,564	$ 695,564
Loader - 11.5 m3	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Loader - 17 m3	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Loader - 21 m3 (L-1350)	$ 5,042,842	$ 5,042,842	$ 1,008,568	$ 1,008,568	$ 1,008,568	$ 1,008,568	$ 1,008,568	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Hydraulic Shovel - 17 m3 O&K 120C	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Hydraulic Shovel - 21 m3 O&K 170C	$ 9,158,265	$ 18,316,530	$ -	$ 1,831,653	$ 3,663,306	$ 3,663,306	$ 3,663,306	$ 3,663,306	$ 1,831,653	$ -	$ -	$ -	$ -	$ -
Hydraulic Shovel - 28 m3 O&K 200C	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Hydraulic Shovel - 34 m3 O&K 340C	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Breaker Loader - 6.5 cubic metre	$ 1,159,274	$ 2,318,548	$ -	$ -	$ 231,855	$ 231,855	$ 231,855	$ 231,855	$ 231,855	$ 231,855	$ 231,855	$ 231,855	$ 231,855	$ 231,855
	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Haulage Trucks (240 ton)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Haulage Trucks (200 ton)	$ 3,709,677	$ 48,225,801	$ 2,967,742	$ 3,709,677	$ 7,419,354	$ 8,903,225	$ 9,645,160	$ 6,677,419	$ 5,935,483	$ 2,225,806	$ 741,935	$ -	$ -	$ -
Haulage Trucks (150 ton)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Haulage Trucks (100 ton)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Tracked Dozer (433 kW)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Tracked Dozer (306 kW)	$ 1,275,201	$ 6,376,007	$ 510,081	$ 510,081	$ 765,121	$ 765,121	$ 765,121	$ 255,040	$ 255,040	$ 510,081	$ 510,081	$ 510,081	$ 510,081	$ 510,081
Tracked Dozer (231 kW)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Grader (233 kW)	$ 927,419	$ 3,709,677	$ 370,968	$ 370,968	$ 370,968	$ 370,968	$ 370,968	$ -	$ -	$ 370,968	$ 370,968	$ 370,968	$ 370,968	$ 370,968
Rubber Tired Dozer (350 kW)	$ 1,507,056	$ 3,014,113	$ 301,411	$ 301,411	$ 301,411	$ 301,411	$ 301,411	$ -	$ -	$ 301,411	$ 301,411	$ 301,411	$ 301,411	$ 301,411
Utility Backhoe with hammer (2.3 cubic metre)	$ 585,433	$ 585,433	$ 117,087	$ 117,087	$ 117,087	$ 117,087	$ 117,087	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Water Truck (Sterling)	$ 336,189	$ 672,379	$ 134,476	$ 134,476	$ 134,476	$ 134,476	$ 134,476	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Tool Carrier	$ 405,746	$ 405,746	$ 81,149	$ 81,149	$ 81,149	$ 81,149	$ 81,149	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Blasting Skid Steer Loader	$ 68,959	$ 275,834	$ 68,959	$ 68,959	$ -	$ -	$ -	$ 68,959	$ 68,959	$ -	$ -	$ -	$ -	$ -
Light Plants	$ 18,035	$ 378,741	$ 63,124	$ 63,124	$ -	$ -	$ 63,124	$ 63,124	$ -	$ -	$ 63,124	$ 63,124	$ -	$ -
Lube/Fuel Truck	$ 359,375	$ 359,375	$ 71,875	$ 71,875	$ 71,875	$ 71,875	$ 71,875	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Mechanics Truck	$ 244,007	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Welding Truck	$ 233,398	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Crewcab Pickups	$ 55,167	$ 662,002	$ 55,167	$ 55,167	$ 55,167	$ 55,167	$ 55,167	$ 55,167	$ 55,167	$ 55,167	$ 55,167	$ 55,167	$ 55,167	$ 55,167
Blasters Truck	$ 55,167	$ 165,500	$ 27,583	$ 27,583	$ -	$ -	$ -	$ 27,583	$ 27,583	$ -	$ -	$ -	$ 27,583	$ 27,583
Pumps	$ 47,741	$ 477,405	$ 47,741	$ 47,741	$ 47,741	$ 47,741	$ -	$ 47,741	$ 47,741	$ 47,741	$ 47,741	$ -	$ 47,741	$ 47,741
Pickup Truck	$ 48,801	$ 585,617	$ 48,801	$ 48,801	$ 48,801	$ 48,801	$ 48,801	$ 48,801	$ 48,801	$ 48,801	$ 48,801	$ 48,801	$ 48,801	$ 48,801
Manbus	$ 84,872	$ 254,616	$ 42,436	$ 42,436	$ -	$ -	$ -	$ 42,436	$ 42,436	$ -	$ -	$ -	$ 42,436	$ 42,436
Ambulance	$ 106,090	$ 106,090	$ 53,045	$ 53,045	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Fire Truck	$ 275,834	$ 275,834	$ 137,917	$ 137,917	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Compactor	$ 301,411	$ 301,411	$ 60,282	$ 60,282	$ 60,282	$ 60,282	$ 60,282	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Lowboy	$ 106,090	$ 106,090	$ 53,045	$ 53,045	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -

Spanish Mountain Gold

Spanish Mountain Project

Project Number: 09SPAN0100

Capital and Operating Cost Estimates - 31 October 2010

Tonnage Option		25ktpd		40ktpd
Capital Costs				
Directs	$	152,643,468	$	212,892,264
Contingency	$	31,544,644	$	53,701,547
Indirects	$	32,744,198	$	45,765,485
Total	$	216,932,311	$	312,359,296
Operating Costs				
Fixed Costs	$	1.48	$	1.15
Variable	$	3.85	$	3.85
Total	$	5.33	$	5.00

Spanish Mountain Gold

Spanish Mountain

G&A Calculation

	Mill Production Options (tonnes per day)			
	25,000		40,000	
	Dollars	$/tonne	Dollars	$/tonne
Salaried Staff	$ 879,000	$ 0.10	$ 1,218,000	$ 0.08
Hourly Personnel	$ 1,645,000	$ 0.18	$ 2,246,000	$ 0.16
Site Operation and Maintenace Supplies	$ 115,000	$ 0.01	$ 150,000	$ 0.01
Site Power	$ 100,000	$ 0.01	$ 100,000	$ 0.01
Williams Lake Office	$ 100,000	$ 0.01	$ 100,000	$ 0.01
Information Systems (Hardware/Software)	$ 100,000	$ 0.01	$ 100,000	$ 0.01
Communications	$ 100,000	$ 0.01	$ 300,000	$ 0.02
Public/Community Relations	$ 100,000	$ 0.01	$ 100,000	$ 0.01
Recruitment and Training	$ 125,000	$ 0.01	$ 200,000	$ 0.01
Safety and Medical Supplies	$ 55,000	$ 0.01	$ 70,000	$ 0.00
Consultants	$ 170,000	$ 0.02	$ 230,000	$ 0.02
Legal and Audit Fees	$ 150,000	$ 0.02	$ 150,000	$ 0.01
Taxes and Insurance	$ -	$ -	$ -	$ -
Logistics	$ 120,000	$ 0.01	$ 180,000	$ 0.01
Office Supplies	$ 75,000	$ 0.01	$ 75,000	$ 0.01
Environmental Monitoring	$ 250,000	$ 0.03	$ 250,000	$ 0.02
Subtotal	$ 4,084,000	$ 0.45	$ 5,469,000	$ 0.38
Sustaining Capital @ 5% of Operating	$ -	$ -	$ -	$ -
Total G&A	$ 4,084,000	**$ 0.45**	$ 5,469,000	**$ 0.38**
			9	

		Mill Production Options (tonnes per day)			
		25,000		40,000	
Salaried Staff	**Annual Rate**	**Dollars**	**Employees**	**Dollars**	**Employees**
General Manager	$ 150,000	$ 150,000	1	$ 150,000	1
Executive Secretary	$ 65,000	$ 65,000	1	$ 65,000	1
Assistant General Manager	$ 130,000	$ -	0	$ -	0
Human Resources Manager	$ 85,000	$ -	0	$ 85,000	1
Office Manager	$ 75,000	$ 75,000	1	$ 75,000	1
Warehouse Supervisor	$ 75,000	$ 75,000	1	$ 75,000	1
Purchasing Supervisor	$ 75,000	$ 75,000	1	$ 75,000	1
Buyer	$ 65,000	$ 65,000	1	$ 130,000	2
Environmental Manager	$ 85,000	$ 85,000	1	$ 85,000	1
Computer Technicians	$ 55,000	$ 55,000	1	$ 110,000	2
Safety and Training Officer	$ 70,000	$ 70,000	1	$ 140,000	2
Security Manager	$ 85,000	$ -	0	$ -	0
Camp Administrator	$ -	$ -	0	$ -	0
Subtotal		$ 715,000		$ 990,000	
Staff Burden	23%	$ 164,450		$ 227,700	
Total Salaried Staff		$ 879,450		$ 1,217,700	
Hourly Personnel		**Dollars**	**Employees**	**Dollars**	**Employees**
Accountant	$ 66,600	$ 66,600	1	$ 133,200	2
Payroll Clerk	$ 56,200	$ 56,200	1	$ 112,400	2
Accounts Clerk	$ 56,200	$ 56,200	1	$ 112,400	2
Secretary	$ 50,000	$ 100,000	2	$ 200,000	4
Warehouse Attendent	$ 55,100	$ -	0	$ -	0
Site Maintenance	$ 60,400	$ 60,400	1	$ 60,400	1
Environmental Technician	$ 64,600	$ 258,400	4	$ 258,400	4
Human Resources Clerk	$ 56,200	$ 56,200	1	$ 112,400	2
Security Supervisors	$ 65,000	$ 65,000	1	$ 65,000	1
Security Officers	$ 64,000	$ 512,000	8	$ 512,000	8
First Aid	$ 60,000	$ -	0	$ 60,000	1
Janitors	$ 42,600	$ 85,200	2	$ 170,400	4
Subtotal		$ 1,316,200		$ 1,796,600	
Hourly Burden	25%	$ 329,050		$ 449,150	
Total Hourly Personnel		$ 1,645,250		$ 2,245,750	

TMF Operating Cost Estimate

			Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
TMF Operating Cost															
	Processed Ore	tonnes	116,905,974			10,800,000	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	12,124,406	10,105,983	9,193,317	2,682,268
	Tailings Operating Cost	$	$ 13,410,000			$ 1,341,000	$ 1,341,000	$ 1,341,000	$ 1,341,000	$ 1,341,000	$ 1,341,000	$ 1,341,000	$ 1,341,000	$ 1,341,000	$ 1,341,000
	Tailings Operating Unit Cost	$/tonne	$ 0.11			$ 0.12	$ 0.09	$ 0.09	$ 0.09	$ 0.09	$ 0.09	$ 0.11	$ 0.13	$ 0.15	$ 0.50

SPANISH MOUNTAIN GOLD LTD.
SPANISH MOUNTAIN PROJECT

CONCEPT DEVELOPMENT STUDY OF WASTE MANAGEMENT FACILITY H2 @ 40,000 TPD
ESTIMATED OPERATING COSTS

Print Feb/24/09 14:55:25

Description	Units	Unit Cost	Year 1 Quantity	Year 1 Cost	Year 2 Quantity	Year 2 Cost	Year 3 Quantity	Year 3 Cost	Year 4 Quantity	Year 4 Cost	Year 5 Quantity	Year 5 Cost	Year 6 Quantity	Year 6 Cost	Year 7 Quantity	Year 7 Cost	Year 8 Quantity	Year 8 Cost	Year 9 Quantity	Year 9 Cost	Year 10 Quantity	Year 10 Cost
		Stage			2		3		4		5		6		7		8		9			
ANNUAL OPERATING COST																						
Service road maintenance	km	$20,000	5.8	$115,000	5.8	$115,000	5.8	$115,000	5.8	$115,000	5.8	$115,000	5.8	$115,000	5.8	$115,000	5.8	$115,000	5.8	$115,000	5.8	$115,000
Embankment maintenance	year	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000
Manpower	year	$80,000	1	$80,000	1	$80,000	1	$80,000	1	$80,000	1	$80,000	1	$80,000	1	$80,000	1	$80,000	1	$80,000	1	$80,000
Power - Reclaim pumping	MWh	$39	14000	$546,000	14000	$546,000	14000	$546,000	14000	$546,000	14000	$546,000	14000	$546,000	14000	$546,000	14000	$546,000	14000	$546,000	14000	$546,000
Reclaim barge relocation	ea	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000	1	$50,000
Environmental compliance	PS	$250,000	1	$250,000	1	$250,000	1	$250,000	1	$250,000	1	$250,000	1	$250,000	1	$250,000	1	$250,000	1	$250,000	1	$250,000
Engineering support and reporting	PS	$250,000	1	$250,000	1	$250,000	1	$250,000	1	$250,000	1	$250,000	1	$250,000	1	$250,000	1	$250,000	1	$250,000	1	$250,000
		Sub-Total		$1,341,000		$1,341,000		$1,341,000		$1,341,000		$1,341,000		$1,341,000		$1,341,000		$1,341,000		$1,341,000		$1,341,000

APPENDIX G

Economic Analysis

Economic Analysis Spreadsheets

COMMODITY PRICE SCENARIOS

		Engineering Base	Financial Base
Gold Price	$US/ounce	$950.00	$1,100.00
Gold Refining Charge	$US/ounce	$8.00	$8.00
Gold Refinery Payable	%	99.5%	99.5%
Net Gold Price	$US/ounce	$937.25	$1,086.50

Exchange Rate	$Cdn:$US	1.10	1.10

Gold Price	$Cdn/ounce	$1,045.00	$1,210.00
Gold Refining Charge	$Cdn/ounce	$8.80	$8.80
Gold Refinery Payable	%	99.5%	99.5%
Net Gold Price	$Cdn/ounce	$1,030.98	$1,195.15

Production Rate Options

		Process Rate - Ore tonnes per day		
		25,000	**40,000**	
Mining Cost - Average Rate	$/tonne	$1.43	$1.26	
Mine Equipment Leasing	Y/N	Y	Y	*Lease Interest Rate*
Mine Leasing Cost - Average Rate	$/tonne	$0.27	$0.28	3%
Total Open Pit Mining Cost - Average	$/tonne	$1.70	$1.54	
Processing Cost	$/tonne ore	$5.33	$5.00	
Tailings Cost	$/tonne ore	$0.16	$0.11	
Total Processing and Tailings cost	$/tonne ore	$5.49	$5.12	
General & Administrative	$/tonne ore	$0.45	$0.38	

Average Insitu Grade	gram/tonne	0.56	0.53
Average Diluted Grade	gram/tonne	0.54	0.51

Owners Cost	$	$8,000,000	$8,000,000

Recovery

Gold		**Gold Grade >**	**Recovery**
Gold Grade	grams/tonne	0.50	90.0%
Gold Grade	grams/tonne	0.30	90.0%
Gold Grade	grams/tonne	0.00	90.0%

Royalty	%	0.0%	Spanish Mountain will be stating that the NPV's are before taxes and royalties

Adjustment Factors

		25,000 tpd		40,000 tpd	
	Capital Dollars	**Indirects**	**Contingency**	**Indirects**	**Contingency**
OP Mining	100%	10.0%	15.0%	10.0%	15.0%
Processing	100%	21.5%	20.7%	21.5%	25.2%
Infrastructure	100%	28.4%	18.5%	27.6%	18.1%
Environmental	100%	0.0%	20.0%	0.0%	20.0%
Operating Cost Factor	100%				

Economic Evaluation Results

	Mining Process Rate	Net Present Value ($ millions)			Internal	Net Revenue	Total Operating	Capital Cost ($ Millions)							
Metal Price Option	**tonnes/day**	**0%**	**5%**	**8%**	**Rate of Return**	**less Royalty**	**Cost**	**Total**	**OP Mining**	**UG Mining**	**Processing**	**Infrastructure**	**Environmental**	**Indirects**	**Contingency**
Engineering Base	25,000	$133	$14	-$32	5.8%	**$1,441**	**$954**	**$353.1**	$0.0	$0.0	$154.2	$78.0	$17.5	$55.2	$48.3
	40,000	$128	$4	-$47	5.2%	**$1,777**	**$1,185**	**$463.4**	$0.0	$0.0	$215.0	$87.4	$18.5	$70.4	$72.1
Financial Base	25,000	$363	$174	$99	14.0%	**$1,670**	**$954**	**$353.1**	$0.0	$0.0	$154.2	$78.0	$17.5	$55.2	$48.3
	40,000	$411	$209	$125	14.7%	**$2,060**	**$1,185**	**$463.4**	$0.0	$0.0	$215.0	$87.4	$18.5	$70.4	$72.1

No Leasing

COMMODITY PRICE SCENARIOS

		Engineering Base	Financial Base
Gold Price	$US/ounce	$950.00	$1,100.00
Gold Refining Charge	$US/ounce	$8.00	$8.00
Gold Refinery Payable	%	99.5%	99.5%
Net Gold Price	$US/ounce	$937.25	$1,086.50
Exchange Rate	$Cdn:$US	1.10	1.10
Gold Price	$Cdn/ounce	$1,045.00	$1,210.00
Gold Refining Charge	$Cdn/ounce	$8.80	$8.80
Gold Refinery Payable	%	99.5%	99.5%
Net Gold Price	$Cdn/ounce	$1,030.98	$1,195.15

Production Rate Options

		Process Rate - Ore tonnes per day		
		25,000	**40,000**	
Mining Cost - Average Rate	$/tonne	$1.43	$1.26	
Mine Equipment Leasing	Y/N	N	N	*Lease Interest Rate*
Mine Leasing Cost - Average Rate	$/tonne	$0.00	$0.00	3%
Total Open Pit Mining Cost - Average	$/tonne	$1.43	$1.26	
Processing Cost	$/tonne ore	$5.33	$5.00	
Tailings Cost	$/tonne ore	$0.16	$0.11	
Total Processing and Tailings cost	$/tonne ore	$5.49	$5.12	
General & Administrative	$/tonne ore	$0.45	$0.38	
Average Insitu Grade	gram/tonne	0.56	0.53	
Average Diluted Grade	gram/tonne	0.54	0.51	
Owners Cost	$	$8,000,000	$8,000,000	

Recovery

Gold		**Gold Grade >**	**Recovery**
Gold Grade	grams/tonne	0.50	90.0%
Gold Grade	grams/tonne	0.30	90.0%
Gold Grade	grams/tonne	0.00	90.0%
Royalty	%	0.0%	Spanish Mountain will be stating that the NPV's are before taxes and royalties

Adjustment Factors

		25,000 tpd		40,000 tpd	
	Capital Dollars	**Indirects**	**Contingency**	**Indirects**	**Contingency**
OP Mining	100%	10.0%	15.0%	10.0%	15.0%
Processing	100%	21.5%	20.7%	21.5%	25.2%
Infrastructure	100%	28.4%	18.5%	27.6%	18.1%
Environmental	100%	0.0%	20.0%	0.0%	20.0%
Operating Cost Factor	100%				

Economic Evaluation Results

	Mining Process Rate	Net Present Value ($ millions)			Internal	Net Revenue	Total Operating	Capital Cost ($ Millions)							
Metal Price Option	**tonnes/day**	**0%**	**5%**	**8%**	**Rate of Return**	**less Royalty**	**Cost**	**Total**	**OP Mining**	**UG Mining**	**Processing**	**Infrastructure**	**Environmental**	**Indirects**	**Contingency**
Engineering Base	25,000	$127	$3	-$46	5.2%	**$1,441**	**$888**	**$426.0**	$58.3	$0.0	$154.2	$78.0	$17.5	$61.0	$57.0
	40,000	$121	-$12	-$66	4.4%	**$1,777**	**$1,086**	**$570.6**	$85.7	$0.0	$215.0	$87.4	$18.5	$78.9	$85.0
Financial Base	25,000	$357	$163	$86	12.8%	**$1,670**	**$888**	**$426.0**	$58.3	$0.0	$154.2	$78.0	$17.5	$61.0	$57.0
	40,000	$404	$193	$106	13.2%	**$2,060**	**$1,086**	**$570.6**	$85.7	$0.0	$215.0	$87.4	$18.5	$78.9	$85.0

Leasing with $US1,200 Gold Price

COMMODITY PRICE SCENARIOS

		Engineering Base	Financial Base
Gold Price	$US/ounce	$950.00	$1,200.00
Gold Refining Charge	$US/ounce	$8.00	$8.00
Gold Refinery Payable	%	99.5%	99.5%
Net Gold Price	$US/ounce	$937.25	$1,186.00
Exchange Rate	$Cdn:$US	1.10	1.10
Gold Price	$Cdn/ounce	$1,045.00	$1,320.00
Gold Refining Charge	$Cdn/ounce	$8.80	$8.80
Gold Refinery Payable	%	99.5%	99.5%
Net Gold Price	$Cdn/ounce	$1,030.98	$1,304.60

Production Rate Options

		Process Rate - Ore tonnes per day		
		25,000	**40,000**	
Mining Cost - Average Rate	$/tonne	$1.43	$1.26	
Mine Equipment Leasing	Y/N	Y	Y	*Lease Interest Rate*
Mine Leasing Cost - Average Rate	$/tonne	$0.27	$0.28	3%
Total Open Pit Mining Cost - Average	$/tonne	$1.70	$1.54	
Processing Cost	$/tonne ore	$5.33	$5.00	
Tailings Cost	$/tonne ore	$0.16	$0.11	
Total Processing and Tailings cost	$/tonne ore	$5.49	$5.12	
General & Administrative	$/tonne ore	$0.45	$0.38	
Average Insitu Grade	gram/tonne	0.56	0.53	
Average Diluted Grade	gram/tonne	0.54	0.51	
Owners Cost	$	$8,000,000	$8,000,000	

Recovery

Gold		**Gold Grade >**	**Recovery**
Gold Grade	grams/tonne	0.50	90.0%
Gold Grade	grams/tonne	0.30	90.0%
Gold Grade	grams/tonne	0.00	90.0%

Royalty	%	0.0%	Spanish Mountain will be stating that the NPV's are before taxes and royalties

Adjustment Factors

		25,000 tpd		40,000 tpd	
	Capital Dollars	**Indirects**	**Contingency**	**Indirects**	**Contingency**
OP Mining	100%	10.0%	15.0%	10.0%	15.0%
Processing	100%	21.5%	20.7%	21.5%	25.2%
Infrastructure	100%	28.4%	18.5%	27.6%	18.1%
Environmental	100%	0.0%	20.0%	0.0%	20.0%
Operating Cost Factor	100%				

Economic Evaluation Results

	Mining Process Rate	Net Present Value ($ millions)			Internal	Net Revenue	Total Operating	Capital Cost ($ Millions)							
Metal Price Option	**tonnes/day**	**0%**	**5%**	**8%**	**Rate of Return**	**less Royalty**	**Cost**	**Total**	**OP Mining**	**UG Mining**	**Processing**	**Infrastructure**	**Environmental**	**Indirects**	**Contingency**
Engineering Base	25,000	$133	$14	-$32	5.8%	**$1,441**	**$954**	**$353.1**	$0.0	$0.0	$154.2	$78.0	$17.5	$55.2	$48.3
	40,000	$128	$4	-$47	5.2%	**$1,777**	**$1,185**	**$463.4**	$0.0	$0.0	$215.0	$87.4	$18.5	$70.4	$72.1
Financial Base	25,000	$516	$281	$187	18.7%	**$1,823**	**$954**	**$353.1**	$0.0	$0.0	$154.2	$78.0	$17.5	$55.2	$48.3
	40,000	$600	$346	$240	20.2%	**$2,249**	**$1,185**	**$463.4**	$0.0	$0.0	$215.0	$87.4	$18.5	$70.4	$72.1

Leasing with $US1300 Gold

COMMODITY PRICE SCENARIOS

		Engineering Base	Financial Base
Gold Price	$US/ounce	$950.00	$1,300.00
Gold Refining Charge	$US/ounce	$8.00	$8.00
Gold Refinery Payable	%	99.5%	99.5%
Net Gold Price	$US/ounce	$937.25	$1,285.50

Exchange Rate	$Cdn:$US	1.10	1.10

Gold Price	$Cdn/ounce	$1,045.00	$1,430.00
Gold Refining Charge	$Cdn/ounce	$8.80	$8.80
Gold Refinery Payable	%	99.5%	99.5%
Net Gold Price	$Cdn/ounce	$1,030.98	$1,414.05

Production Rate Options

		Process Rate - Ore tonnes per day		
		25,000	**40,000**	
Mining Cost - Average Rate	$/tonne	$1.43	$1.26	
Mine Equipment Leasing	Y/N	Y	Y	*Lease Interest Rate*
Mine Leasing Cost - Average Rate	$/tonne	$0.27	$0.28	3%
Total Open Pit Mining Cost - Average	$/tonne	$1.70	$1.54	
Processing Cost	$/tonne ore	$5.33	$5.00	
Tailings Cost	$/tonne ore	$0.16	$0.11	
Total Processing and Tailings cost	$/tonne ore	$5.49	$5.12	
General & Administrative	$/tonne ore	$0.45	$0.38	

Average Insitu Grade	gram/tonne	0.56	0.53
Average Diluted Grade	gram/tonne	0.54	0.51

Owners Cost	$	$8,000,000	$8,000,000

Recovery

Gold		**Gold Grade >**	**Recovery**
Gold Grade	grams/tonne	0.50	90.0%
Gold Grade	grams/tonne	0.30	90.0%
Gold Grade	grams/tonne	0.00	90.0%

Royalty	%	0.0%	Spanish Mountain will be stating that the NPV's are before taxes and royalties

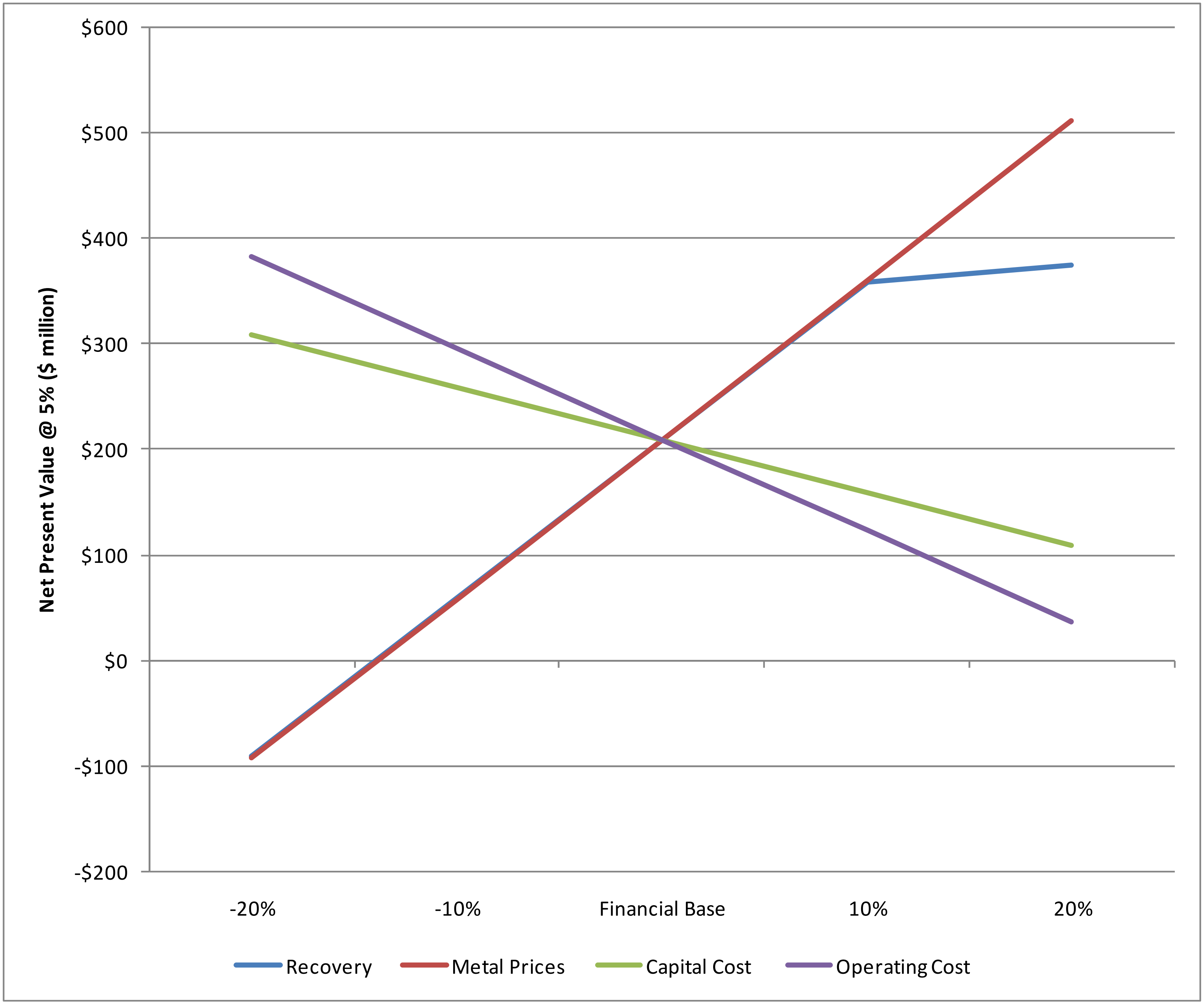
Adjustment Factors

		25,000 tpd		40,000 tpd	
	Capital Dollars	**Indirects**	**Contingency**	**Indirects**	**Contingency**
OP Mining	100%	10.0%	15.0%	10.0%	15.0%
Processing	100%	21.5%	20.7%	21.5%	25.2%
Infrastructure	100%	28.4%	18.5%	27.6%	18.1%
Environmental	100%	0.0%	20.0%	0.0%	20.0%
Operating Cost Factor	100%				

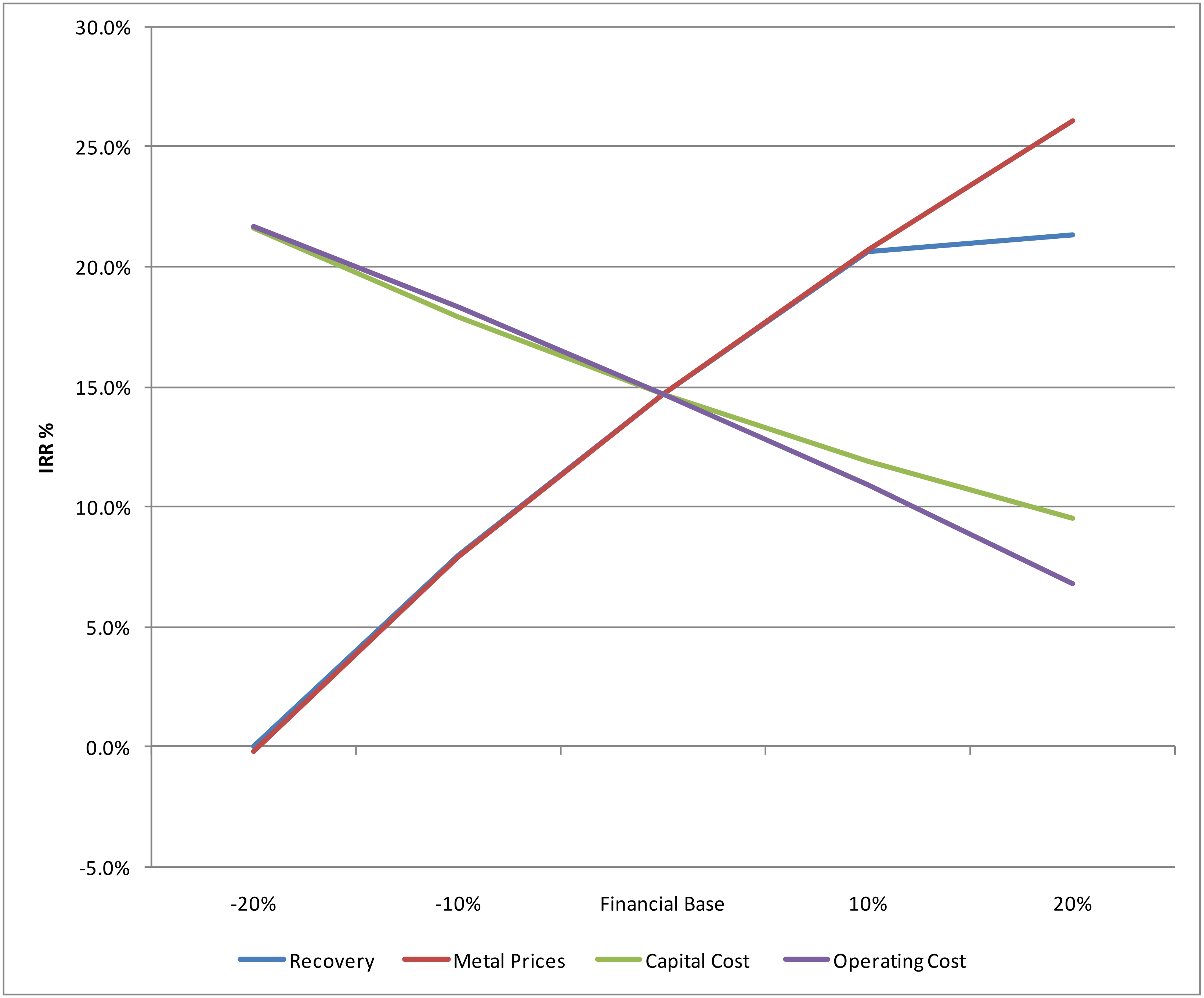
Economic Evaluation Results

	Mining Process Rate	Net Present Value ($ millions)			Internal	Net Revenue	Total Operating	Capital Cost ($ Millions)							
Metal Price Option	**tonnes/day**	**0%**	**5%**	**8%**	**Rate of Return**	**less Royalty**	**Cost**	**Total**	**OP Mining**	**UG Mining**	**Processing**	**Infrastructure**	**Environmental**	**Indirects**	**Contingency**
Engineering Base	25,000	$133	$14	-$32	5.8%	**$1,441**	**$954**	**$353.1**	$0.0	$0.0	$154.2	$78.0	$17.5	$55.2	$48.3
	40,000	$128	$4	-$47	5.2%	**$1,777**	**$1,185**	**$463.4**	$0.0	$0.0	$215.0	$87.4	$18.5	$70.4	$72.1
Financial Base	25,000	$669	$388	$275	23.1%	**$1,976**	**$954**	**$353.1**	$0.0	$0.0	$154.2	$78.0	$17.5	$55.2	$48.3
	40,000	$788	$483	$355	25.2%	**$2,437**	**$1,185**	**$463.4**	$0.0	$0.0	$215.0	$87.4	$18.5	$70.4	$72.1

Spider Graph at $US1100 Gold Base



Spider Graph at $US1100 Gold Base



40K Cashflow - Engineering Base Price $US950 Gold

Engineering Base

Plant Throughput	40,000	tonnes per day													
		Total		Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Mill Production															
Mill Feed	tonnes	116,905,974		-	-	10,800,000	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	12,124,406	10,105,983	9,193,317	2,682,268
Gold Grade	g/t	0.51		-	-	0.55	0.63	0.53	0.50	0.49	0.49	0.45	0.42	0.49	0.54
Gold Recovery	%			0.0%	0.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%
Insitu Gold	ounces	1,915,177		-	-	192,597	290,288	246,880	229,465	228,326	225,269	173,730	138,026	144,346	46,251
Recovered Gold	ounces	1,723,660		-	-	173,337	261,259	222,192	206,519	205,494	202,742	156,357	124,224	129,911	41,626
Mine Production															
Open Pit															
Ore															
Ore to Mill	tonnes	111,327,446		-	-	6,875,081	12,962,213	14,184,178	14,400,000	14,400,000	14,400,000	12,124,406	10,105,983	9,193,317	2,682,268
Ore to Stockpile	tonnes	5,578,528		1,646,642	3,931,886	-	-	-	-	-	-	-	-	-	-
Stockpile to Mill	tonnes	5,578,528		-	-	3,924,919	1,437,787	215,822	-	-	-	-	-	-	-
Waste	tonnes	230,126,161		4,088,038	7,590,346	13,272,973	30,896,338	32,744,747	31,067,578	31,682,172	21,054,746	21,002,742	21,785,538	12,915,230	2,025,713
Total Material	tonnes	352,610,663		5,734,680	11,522,232	24,072,973	45,296,338	47,144,747	45,467,578	46,082,172	35,454,746	33,127,148	31,891,521	22,108,547	4,707,981
Strip Ratio		1.97		-	-	1.23	2.15	2.27	2.16	2.20	1.46	1.73	2.16	1.40	0.76
Operating Cost															
Open Pit Mining	dollars	437,926,684	3.75	15,398,925	21,659,052	25,721,770	43,679,699	50,341,456	53,036,559	52,306,462	47,065,529	43,488,447	42,922,544	34,389,678	7,916,562
Leasing Cost	dollars	99,571,237	0.85	6,569,238	9,142,826	15,072,725	16,556,596	17,313,915	11,529,212	8,892,500	4,487,394	3,066,646	2,276,971	2,331,607	2,331,607
Processing	dollars	598,074,547	5.12	0	0	55,251,283	73,668,378	73,668,378	73,668,378	73,668,378	73,668,378	62,026,759	51,700,790	47,031,719	13,722,106
G&A	dollars	49,868,915	0.43	2,734,500	2,734,500	4,101,750	5,469,000	5,469,000	5,469,000	5,469,000	5,469,000	4,604,748	3,838,168	3,491,545	1,018,703
Subtotal Operating	dollars	1,185,441,383		24,702,663	33,536,379	100,147,529	139,373,674	146,792,749	143,703,149	140,336,340	130,690,301	113,186,601	100,738,473	87,244,549	24,988,978
Unit Cost															
Operating Cost per tonne milled	$/tonne ore	$ 10.14		$ -	$ -	$ 9.27	$ 9.68	$ 10.19	$ 9.98	$ 9.75	$ 9.08	$ 9.34	$ 9.97	$ 9.49	$ 9.32
Mining Cost per Tonne Milled	$/tonne ore	$ 3.75		$ -	$ -	$ 2.38	$ 3.03	$ 3.50	$ 3.68	$ 3.63	$ 3.27	$ 3.59	$ 4.25	$ 3.74	$ 2.95
Capital Cost															
Open Pit Mining	dollars	0		0	0	0	0	0	0	0	0	0	0	0	0
Processing	dollars	215,029,000		63,870,000	106,450,000	42,580,000	0	212,900	212,900	425,800	425,800	425,800	212,900	212,900	0
Infrastructure	dollars	87,448,000		47,328,258	29,750,743	1,235,380	955,010	1,295,120	1,219,320	1,537,110	1,134,740	1,609,670	822,560	560,090	0
Environment Costs	dollars	18,500,000		18,500,000	0	0	0	0	0	0	0	0	0	0	0
Indirect	dollars	70,373,968		25,169,293	32,241,920	9,491,958	362,906	492,149	463,344	584,105	431,204	611,678	312,575	212,836	0
Contingency	dollars	72,093,568		26,190,823	32,678,737	11,033,464	238,754	301,294	304,832	360,292	283,687	379,970	205,641	116,073	0
Subtotal Capital	dollars	463,444,536		181,058,374	201,121,399	64,340,803	1,556,670	2,301,463	2,200,396	2,907,308	2,275,431	3,027,118	1,553,676	1,101,899	0
Revenue (after refining)															
Gold Gross Revenue	dollars	1,801,224,350		-	-	181,137,068	273,015,496	232,190,901	215,811,900	214,741,003	211,865,349	163,393,106	129,813,596	135,757,208	43,498,720
less Gold Refining	dollars	15,168,205		-	-	1,525,365	2,299,078	1,955,292	1,817,363	1,808,345	1,784,129	1,375,942	1,093,167	1,143,219	366,305
less Gold Payables	dollars	9,006,122		-	-	905,685	1,365,077	1,160,955	1,079,060	1,073,705	1,059,327	816,966	649,068	678,786	217,494
Subtotal		1,777,050,023		-	-	178,706,018	269,351,341	229,074,655	212,915,477	211,858,953	209,021,893	161,200,199	128,071,361	133,935,204	42,914,922
less Royalty	dollars	0		-	-	-	-	-	-	-	-	-	-	-	-
Net Revenue	dollars	1,777,050,023		-	-	178,706,018	269,351,341	229,074,655	212,915,477	211,858,953	209,021,893	161,200,199	128,071,361	133,935,204	42,914,922
Cashflow															
Operating Cost	dollars	1,185,442,000		24,703,000	33,536,000	100,148,000	139,374,000	146,793,000	143,703,000	140,336,000	130,690,000	113,187,000	100,738,000	87,245,000	24,989,000
Capital Cost	dollars	463,443,000		181,058,000	201,121,000	64,341,000	1,557,000	2,301,000	2,200,000	2,907,000	2,275,000	3,027,000	1,554,000	1,102,000	0
Revenue	dollars	1,777,049,000		0	0	178,706,000	269,351,000	229,075,000	212,915,000	211,859,000	209,022,000	161,200,000	128,071,000	133,935,000	42,915,000
Net Cashflow	dollars	128,164,000		-205,761,000	-234,657,000	14,217,000	128,420,000	79,981,000	67,012,000	68,616,000	76,057,000	44,986,000	25,779,000	45,588,000	17,926,000
Cumulative	dollars			-205,761,000	-440,418,000	-426,201,000	-297,781,000	-217,800,000	-150,788,000	-82,172,000	-6,115,000	38,871,000	64,650,000	110,238,000	128,164,000

NPV (millions) @		
	0%	$128
	5%	$4
	8%	-$47
IRR		5.2%

40K Cashflow - Financial Base Price $US1100 Gold

Financial Base

Plant Throughput	**40,000**	**tonnes per day**													
		Total		**Year -2**	**Year -1**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**
Mill Production															
Mill Feed	tonnes	116,905,974		-	-	10,800,000	14,400,000	14,400,000	14,400,000	14,400,000	14,400,000	12,124,406	10,105,983	9,193,317	2,682,268
Gold Grade	g/t	0.51		-	-	0.55	0.63	0.53	0.50	0.49	0.49	0.45	0.42	0.49	0.54
Gold Recovery	%			0.0%	0.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%
Insitu Gold	ounces	1,915,177		-	-	192,597	290,288	246,880	229,465	228,326	225,269	173,730	138,026	144,346	46,251
Recovered Gold	ounces	1,723,660		-	-	173,337	261,259	222,192	206,519	205,494	202,742	156,357	124,224	129,911	41,626
Mine Production															
Open Pit															
Ore															
Ore to Mill	tonnes	111,327,446		-	-	6,875,081	12,962,213	14,184,178	14,400,000	14,400,000	14,400,000	12,124,406	10,105,983	9,193,317	2,682,268
Ore to Stockpile	tonnes	5,578,528		1,646,642	3,931,886	-	-	-	-	-	-	-	-	-	-
Stockpile to Mill	tonnes	5,578,528		-	-	3,924,919	1,437,787	215,822	-	-	-	-	-	-	-
Waste	tonnes	230,126,161		4,088,038	7,590,346	13,272,973	30,896,338	32,744,747	31,067,578	31,682,172	21,054,746	21,002,742	21,785,538	12,915,230	2,025,713
Total Material	tonnes	352,610,663		5,734,680	11,522,232	24,072,973	45,296,338	47,144,747	45,467,578	46,082,172	35,454,746	33,127,148	31,891,521	22,108,547	4,707,981
Strip Ratio		1.97		-	-	1.23	2.15	2.27	2.16	2.20	1.46	1.73	2.16	1.40	0.76
Operating Cost															
Open Pit Mining	dollars	437,926,684	3.75	15,398,925	21,659,052	25,721,770	43,679,699	50,341,456	53,036,559	52,306,462	47,065,529	43,488,447	42,922,544	34,389,678	7,916,562
Leasing Cost	dollars	99,571,237	0.85	6,569,238	9,142,826	15,072,725	16,556,596	17,313,915	11,529,212	8,892,500	4,487,394	3,066,646	2,276,971	2,331,607	2,331,607
Processing	dollars	598,074,547	5.12	0	0	55,251,283	73,668,378	73,668,378	73,668,378	73,668,378	73,668,378	62,026,759	51,700,790	47,031,719	13,722,106
G&A	dollars	49,868,915	0.43	2,734,500	2,734,500	4,101,750	5,469,000	5,469,000	5,469,000	5,469,000	5,469,000	4,604,748	3,838,168	3,491,545	1,018,703
Subtotal Operating	dollars	1,185,441,383	10.14	24,702,663	33,536,379	100,147,529	139,373,674	146,792,749	143,703,149	140,336,340	130,690,301	113,186,601	100,738,473	87,244,549	24,988,978
			688							627					
Unit Cost															
Operating Cost per tonne milled	$/tonne ore	$ 10.14		$ -	$ -	$ 9.27	$ 9.68	$ 10.19	$ 9.98	$ 9.75	$ 9.08	$ 9.34	$ 9.97	$ 9.49	$ 9.32
Mining Cost per Tonne Milled	$/tonne ore	$ 3.75		$ -	$ -	$ 2.38	$ 3.03	$ 3.50	$ 3.68	$ 3.63	$ 3.27	$ 3.59	$ 4.25	$ 3.74	$ 2.95
Capital Cost															
Open Pit Mining	dollars	0		0	0	0	0	0	0	0	0	0	0	0	0
Processing	dollars	215,029,000		63,870,000	106,450,000	42,580,000	0	212,900	212,900	425,800	425,800	425,800	212,900	212,900	0
Infrastructure	dollars	87,448,000		47,328,258	29,750,743	1,235,380	955,010	1,295,120	1,219,320	1,537,110	1,134,740	1,609,670	822,560	560,090	0
Environment Costs	dollars	18,500,000		18,500,000	0	0	0	0	0	0	0	0	0	0	0
Indirect	dollars	70,373,968		25,169,293	32,241,920	9,491,958	362,906	492,149	463,344	584,105	431,204	611,678	312,575	212,836	0
Contingency	dollars	72,093,568		26,190,823	32,678,737	11,033,464	238,754	301,294	304,832	360,292	283,687	379,970	205,641	116,073	0
Subtotal Capital	dollars	463,444,536		181,058,374	201,121,399	64,340,803	1,556,670	2,301,463	2,200,396	2,907,308	2,275,431	3,027,118	1,553,676	1,101,899	0
Revenue (after refining)															
Gold Gross Revenue	dollars	2,085,628,195		-	-	209,737,658	316,123,206	268,852,622	249,887,463	248,647,478	245,317,773	189,192,018	150,310,480	157,192,557	50,366,940
less Gold Refining	dollars	15,168,205		-	-	1,525,365	2,299,078	1,955,292	1,817,363	1,808,345	1,784,129	1,375,942	1,093,167	1,143,219	366,305
less Gold Payables	dollars	10,428,141		-	-	1,048,688	1,580,616	1,344,263	1,249,437	1,243,237	1,226,589	945,960	751,552	785,963	251,835
Subtotal		2,060,031,849		-	-	207,163,605	312,243,512	265,553,068	246,820,663	245,595,895	242,307,055	186,870,116	148,465,760	155,263,376	49,748,800
less Royalty	dollars	0		-	-	-	-	-	-	-	-	-	-	-	-
Net Revenue	dollars	2,060,031,849		-	-	207,163,605	312,243,512	265,553,068	246,820,663	245,595,895	242,307,055	186,870,116	148,465,760	155,263,376	49,748,800
Cashflow															
Operating Cost	dollars	1,185,442,000		24,703,000	33,536,000	100,148,000	139,374,000	146,793,000	143,703,000	140,336,000	130,690,000	113,187,000	100,738,000	87,245,000	24,989,000
Capital Cost	dollars	463,443,000		181,058,000	201,121,000	64,341,000	1,557,000	2,301,000	2,200,000	2,907,000	2,275,000	3,027,000	1,554,000	1,102,000	0
Revenue	dollars	2,060,033,000		0	0	207,164,000	312,244,000	265,553,000	246,821,000	245,596,000	242,307,000	186,870,000	148,466,000	155,263,000	49,749,000
Net Cashflow	dollars	411,148,000		-205,761,000	-234,657,000	42,675,000	171,313,000	116,459,000	100,918,000	102,353,000	109,342,000	70,656,000	46,174,000	66,916,000	24,760,000
Cumulative	dollars			-205,761,000	-440,418,000	-397,743,000	-226,430,000	-109,971,000	-9,053,000	93,300,000	202,642,000	273,298,000	319,472,000	386,388,000	411,148,000
NPV (millions) @	**0%**	$411													
	5%	$209													
	8%	$125													
IRR		14.7%													

Tonnes and Grade for Pits with Different Gold Prices at 40K Tradeoff Study Costs

Spanish Mountain Gold							
Spanish Mountain							
Exchange Rate		1.1	40K Pits at various gold prices				
Economic Cones			File 13 spam.pt3				
	Production Rate	tonnes per day	40,000				
	Market Gold Price	$US/ounce	$ 950.00	$ 975.00	$ 1,000.00	$ 1,050.00	$ 1,100.00
	Net Gold Price	$US/ounce	$ 927.75	$ 952.38	$ 977.00	$ 1,026.25	$ 1,075.50
	Net Gold Price	$US/gram	$ 29.83	$ 30.62	$ 31.41	$ 32.99	$ 34.58
	Net Gold Price	$Cdn/gram	$ 32.81	$ 33.68	$ 34.55	$ 36.29	$ 38.04
Mine Cost	Base Cost	$/tonne (all)	$ 1.54	$ 1.54	$ 1.54	$ 1.54	$ 1.54
Processing Costs		$/tonne (ore)	$ 5.12	$ 5.12	$ 5.12	$ 5.12	$ 5.12
G&A		$/tonne (ore)	$ 0.38	$ 0.38	$ 0.38	$ 0.38	$ 0.38
Total Processing and G&A		$/tonne (ore)	$ 5.50	$ 5.50	$ 5.50	$ 5.50	$ 5.50
Gold Grade Item			Au				
Gold Recovery		%	90%	90%	90%	90%	90%
	Top File 13 surface						
	Pit File 13 Surface		d05	d06	d07	d08	d09
	Partials File	*.out					
	Report File	*.rpt, *.sum					
Calculated Cutoff	Mining	g/t	0.24	0.23	0.23	0.22	0.21
	Milling	g/t	0.186	0.181	0.177	0.168	0.161
Mining Cutoff	Cutoff item	value per block	VLB2				
Milling Cutoff		value per tonne	VLT2				
	Ore						
1	Mill Cutoff to 0.4 g/t	tonnes	86,948,854	96,160,998	100,860,383	113,242,402	126,078,187
		g/t	0.27	0.27	0.26	0.26	0.25
2	0.4 - 0.5 g/t	tonnes	19,383,918	20,103,243	20,281,511	20,744,381	21,373,008
		g/t	0.45	0.45	0.45	0.45	0.45
3	0.5 - 0.6 g/t	tonnes	13,335,911	13,867,586	13,942,646	14,183,463	14,533,743
		g/t	0.55	0.55	0.55	0.55	0.55
4	+0.6 g/t	tonnes	38,919,414	40,398,721	40,689,578	41,268,166	41,818,606
		g/t	1.00	1.00	1.00	0.99	0.99
	Total Ore	tonnes	158,588,097	170,530,548	175,774,118	189,438,412	203,803,544
		g/t	0.49	0.48	0.48	0.46	0.44
	Waste	tonnes	278,324,445	301,901,483	304,484,740	310,170,851	321,347,379
	Strip Ratio		1.76	1.77	1.73	1.64	1.58
	Insitu Gold	ounces	2,522,555	2,651,490	2,694,198	2,799,474	2,913,379
	Recovered Gold	ounces	2,270,300	2,386,341	2,424,778	2,519,527	2,622,041
	Net Value	$	$ 771,778,892	$ 834,455,622	$ 899,508,802	$ 1,032,873,583	$ 1,172,302,383
		$/tonne ore	$ 4.87	$ 4.89	$ 5.12	$ 5.45	$ 5.75
	Net Value for Various Gold Prices						
	$ 950.00	$US/ounce	$ 771,778,892	$ 769,816,688	$ 768,148,663	$ 759,887,433	$ 746,162,218
	$ 975.00	$US/ounce	$ 833,274,604	$ 834,455,622	$ 833,828,733	$ 828,133,971	$ 817,185,579
	$ 1,000.00	$US/ounce	$ 894,770,315	$ 899,094,555	$ 899,508,802	$ 896,380,508	$ 888,208,940
	$ 1,050.00	$US/ounce	$ 1,017,761,739	$ 1,028,372,423	$ 1,030,868,941	$ 1,032,873,583	$ 1,030,255,662
	$ 1,100.00	$US/ounce	$ 1,140,753,162	$ 1,157,650,290	$ 1,162,229,080	$ 1,169,366,659	$ 1,172,302,383
	Net Value Per Tonne Ore						
	$ 950.00	$US/ounce	$ 4.87	$ 4.51	$ 4.37	$ 4.01	$ 3.66
	$ 975.00	$US/ounce	$ 5.25	$ 4.89	$ 4.74	$ 4.37	$ 4.01
	$ 1,000.00	$US/ounce	$ 5.64	$ 5.27	$ 5.12	$ 4.73	$ 4.36
	$ 1,050.00	$US/ounce	$ 6.42	$ 6.03	$ 5.86	$ 5.45	$ 5.06
	$ 1,100.00	$US/ounce	$ 7.19	$ 6.79	$ 6.61	$ 6.17	$ 5.75