Report to:

# Technical Report and Preliminary Economic Assessment of the Spanish Mountain Gold Project, Likely, BC

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# TECHNICAL REPORT AND PRELIMINARY ECONOMIC ASSESSMENT OF THE SPANISH MOUNTAIN GOLD PROJECT, LIKELY, BC

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# GLOSSARY

## UNITS OF MEASURE

above mean sea level ........................................ amsl
acre ........................................ ac
ampere ........................................ A
annum (year) ........................................ a

| | |
|---|---|
| bank cubic metres | bcm |
| billion | B |
| billion tonnes | Bt |
| billion years ago | Ga |
| centimetre | cm |
| cubic centimetre | $cm^3$ |
| cubic feet per minute | cfm |
| cubic feet per second | $ft^3/s$ |
| cubic foot | $ft^3$ |
| cubic inch | $in^3$ |
| cubic metre | $m^3$ |
| cubic yard | $yd^3$ |
| Coefficients of Variation | CVs |
| day | d |
| days per week | d/wk |
| days per year (annum) | d/a |
| dead weight tonnes | DWT |
| decibel adjusted | dBa |
| decibel | dB |
| degree | ° |
| degrees Celsius | °C |
| diameter | ø |
| dollar (American) | US$ |
| dollar (Canadian) | CAD$ |
| dry metric ton | dmt |
| foot | ft |
| gallon | gal |
| gallons per minute (US) | gpm |
| gigajoule | GJ |
| gigapascal | GPa |
| gigawatt | GW |
| gram | g |
| grams per litre | g/L |
| grams per tonne | g/t |
| greater than | > |
| hectare (10,000 $m^2$) | ha |
| hertz | Hz |
| horsepower | hp |
| hour | h |
| hours per day | h/d |
| hours per week | h/wk |
| hours per year | h/a |
| inch | " |
| kilo (thousand) | k |
| kilogram | kg |
| kilograms per cubic metre | $kg/m^3$ |

| | |
|---|---|
| kilograms per hour | kg/h |
| kilograms per square metre | $kg/m^2$ |
| kilometre | km |
| kilometres per hour | km/h |
| kilopascal | kPa |
| kilotonne | kt |
| kilovolt | kV |
| kilovolt-ampere | kVA |
| kilovolts | kV |
| kilowatt | kW |
| kilowatt hour | kWh |
| kilowatt hours per tonne (metric ton) | kWh/t |
| kilowatt hours per year | kWh/a |
| less than | < |
| litre | L |
| litres per minute | L/m |
| legabytes per second | Mb/s |
| megapascal | MPa |
| megavolt-ampere | MVA |
| megawatt | MW |
| metre | m |
| metres above sea level | masl |
| metres Baltic sea level | mbsl |
| metres per minute | m/min |
| metres per second | m/s |
| metric ton (tonne) | t |
| microns | µm |
| milligram | mg |
| milligrams per litre | mg/L |
| millilitre | mL |
| millimetre | mm |
| million | M |
| million bank cubic metres | $Mbm^3$ |
| million bank cubic metres per annum | $Mbm^3/a$ |
| million tonnes | Mt |
| minute (plane angle) | ' |
| minute (time) | min |
| month | mo |
| ounce | oz |
| pascal | Pa |
| centipoise | mPa·s |
| parts per million | ppm |
| parts per billion | ppb |
| percent | % |
| pound(s) | lb |
| pounds per square inch | psi |

| | |
|---|---|
| revolutions per minute | rpm |
| second (plane angle) | " |
| second (time) | s |
| specific gravity | SG |
| square centimetre | $cm^2$ |
| square foot | $ft^2$ |
| square inch | $in^2$ |
| square kilometre | $km^2$ |
| square metre | $m^2$ |
| thousand tonnes | kt |
| Three Dimensional | 3D |
| Three Dimensional Model | 3DM |
| tonne (1,000 kg) | t |
| tonnes per day | t/d |
| tonnes per hour | t/h |
| tonnes per year | t/a |
| tonnes seconds per hour metre cubed | $ts/hm^3$ |
| volt | V |
| week | wk |
| weight/weight | w/w |
| wet metric ton | wmt |
| year (annum) | a |

## Abbreviations and Acronyms

| | |
|---|---|
| acid potential | AP |
| acid rock drainage | ARD |
| Acrex Ventures Ltd | Acrex |
| AGP Mining Consultants Inc. | AGP |
| ALS Minerals | ALS |
| Aquarius Resources Ltd | Aquarius Resources |
| arsenic | As |
| Association for the Advancement of Cost Engineering International | AACE |
| atomic absorption spectrophotometer | AAS |
| BC Water Quality Guidelines | BCWQG |
| BGC Engineering Inc. | BGC |
| Bond abrasion index | Ai |
| Bond ball mill work index | BWi |
| Bond crushing work index | CWi |
| Bond rod mill index | RWi |
| calcium | Ca |
| Canadian Council of Ministers of the Environment | CCME |
| Canadian Development Expense | CDE |
| Canadian Environmental Assessment Agency | CEAA |
| Canadian Exploration Expense | CEE |
| Canadian Institute of Mining, Metallurgy and Petroleum | CIM |

| | |
|---|---|
| capital cost estimate | CAPEX |
| carbon | C |
| carbon-in-leach | CIL |
| carboxymethyl cellulose | CMC |
| CDN Resources Laboratories Ltd. | CDN |
| Cedar Mountain Exploration Inc. | Cedar Mountain |
| copper | Cu |
| Cumulative Expenditures Account | CEA |
| cumulative net cash flow | CNCF |
| Cumulative Tax Credit Account | CTCA |
| Cyprus Resources Ltd. | Cyprus Resources |
| Discovery Consultants | Discovery |
| Eco-tech Laboratories Ltd | Eco-tech |
| engineering, procurement and construction management | EPCM |
| Environmental Assessment Office | EAO |
| environmental assessment | EA |
| Extended Gravity Recoverable Gold | EGRG |
| Fisheries and Oceans Canada | DFO |
| Forest Service Road | FSR |
| G&T Metallurgical Services | G&T |
| general and administrative | G&A |
| Giroux Consultants Ltd. | Giroux Consultants |
| global positioning system | GPS |
| gold | Au |
| heating, ventilation and air conditioning | HVAC |
| high-density polyethylene | HDPE |
| Imperial Metals Corporation | Imperial Metals |
| inductively coupled plasma-mass spectrometry | ICP-MS |
| internal rate of return | IRR |
| International Electrotechnical Commission | IEC |
| International Organization for Standardization | ISO |
| investment tax credit | ITC |
| iron | Fe |
| Knelson Research and Technology Centre | Knelson Research |
| Knight Piésold Ltd. | Knight Piésold |
| Lerchs-Grossman | LG |
| Lhtako Dene Nation | Lhtako Dene |
| life-of-mine | LOM |
| metal leaching | ML |
| Metal Mining Effluent Regulations | MMER |
| methyl isobutyl carbinol | MIBC |
| Mineral Titles Online | MTO |
| molybdenum | Mo |
| Moose Mountain Technical Services | MMTS |
| Mt. Calvery Resources Ltd. | Mt. Calvery Resources |
| National Instrument 43-101 | NI 43-101 |

| | |
|---|---|
| Natural Resources Canada | NRCan |
| net cash flow | NCF |
| net present value | NPV |
| net smelter return | NSR |
| neutralization potential | NP |
| non-potentially acid generating | NPAG |
| operating cost estimate | OPEX |
| ordinary kriging | OK |
| Pamicon Developments Ltd | Pamicon |
| payment instead of exploration or development | PIED |
| potassium amyl xanthate | PAX |
| potentially acid generating | PAG |
| preliminary economic assessment | PEA |
| pre-production period 1 | PP 1 |
| pre-production period 2 | PP 2 |
| programmable logic controller | PLC |
| project execution plan | PEP |
| Project Information Centre | e-PIC |
| Qualified Persons | QPs |
| quality assurance/quality control | QA/QC |
| rock quality designation | RQD |
| run-of-mine | ROM |
| SAG Mill Comminution | SMC |
| sediment-hosted vein | SHV |
| semi-autogenous grinding | SAG |
| semi-autogenous-ball milling-crushing | SABC |
| SGS Minerals Services | SGS |
| silver | Ag |
| Skygold Ventures Ltd. | Skygold |
| Soda Creek Indian Band | SCIB |
| sodium cyanide | NaCN |
| sodium hydroxide | NaOH |
| sodium metabisulphite | SMBS |
| Spanish Mountain Gold Ltd. | SMG |
| Spanish Mountain Gold Project | the Project |
| Spanish Mountain Gold Property | the Property |
| SRK Consulting (Canada) Inc. | SRK |
| Standards Council of Canada | SCC |
| Stantec Inc. | Stantec |
| Statement of Work | SOW |
| sulphur | S |
| system impact study | SIS |
| tailings storage facility | TSF |
| Teck Resources | Teck |
| Tetra Tech Wardrop | Tetra Tech |
| total organic carbon | TOC |

| | |
|---|---|
| Universal Transverse Mercator | UTM |
| Voice over Internet Protocol | VoIP |
| waste rock storage areas | WRSAs |
| Williams Lake Indian Band | WLIB |
| x-ray fluorescence spectrometer | XRF |

# 1.0 SUMMARY

## 1.1 INTRODUCTION

Spanish Mountain Gold Ltd. (SMG) retained Tetra Tech to prepare a National Instrument 43-101 (NI 43-101) compliant technical report and preliminary economic assessment (PEA) for the Spanish Mountain Gold Project (the Project). The Spanish Mountain Gold Property (the Property) involves the development of a gold deposit located in southcentral BC, Canada, approximately 6 km southeast of the community of Likely and 66 km northeast of the City of Williams Lake (Figure 1.1).

The Property is situated between Quesnel Lake and Spanish Lake; its centre is located at approximately latitude 52˚ 34' north and longitude 121˚ 28' west. The gold concentrator for the Project has been designed to process a nominal 14,600,000 t/a (or 40,000 t/d) of gold and silver bearing material from an open pit operation, and will produce gold-silver doré as a final product.

The effective date of this report is December 18, 2012 and the effective date of the resource model is August 31, 2012.

General information for the Project is summarized in Table 1.1.

**Table 1.1 General Project Information**

| Description | Unit | Amount |
|---|---|---|
| Estimated Mineral Resources (Measured + Indicated) | Mt | 216 |
| Estimated Mineral Resources (Inferred) | Mt | 316 |
| Life-of-mine (LOM) | years | 14 |
| Milling Rate | t/d | 40,000 |
| Strip Ratio | t waste:mineralized material | 2.29 |
| Total Project Capital Cost | US$ million | 755.9 |
| Average Overall Operating Cost | US$/t milled | 10.68 |
| Net Present Value (NPV) at 5% Discount Rate | US$ million | 454 |
| Gold Price | US$ | 1,462 |
| Internal Rate of Return (IRR) | % | 15 |
| Payback Period | years | 4.4 |

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

**Figure 1.1 Property Location Map**

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

- Tetra Tech – overall project management, mineral processing and metallurgical testing, recovery methods, buildings, access and site roads, market studies and contracts, capital and operating costs, financial analysis, and project execution plan
- Discovery Consultants (Discovery) – property description and location, accessibility, climate, and physiology, history, geological setting and mineralization, deposit types, exploration, drilling, sample preparation, data verification, and adjacent properties
- Giroux Consultants Ltd. (Giroux Consultants) – mineral resource estimate
- Moose Mountain Technical Services (MMTS) – haul roads, mineral reserve estimate and mining
- BGC Engineering Inc. (BGC) – geotechnical engineering for infrastructure, the waste rock storage areas (WRSAs) and the open pit
- Knight Piésold Ltd. (Knight Piésold) – tailings, waste rock and water management, and environmental studies, permits and social or community impact
- Stantec Inc. (Stantec) – power supply to plant site.

## 1.2 PROPERTY DESCRIPTION

The Property is located in the Cariboo region of central BC, 6 km east of the community of Likely, and 66 km northeast of the City of Williams Lake. The Property consists of 46 Mineral Titles Online (MTO) mineral claims, of which 20 are legacy claims. Of the 46 claims, 2 lie on the west side of Quesnel Lake; the other 44 form a contiguous block of claims covering an area of approximately 7,680 ha. The Property is 100% owned by SMG, subject to four separate net smelter return (NSR) royalties on some of the mineral tenures.

The main resource, consisting of the Main and North Zones, is located west of the northwest end of Spanish Lake, and is centred at approximate Universal Transverse Mercator (UTM) coordinates 604425 East and 5827900 North (Datum North American Datum (NAD)83, Zone 10). It is located mainly within mineral claim 204667 and mineral claims 204225 and 204226.

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

## 1.3 GEOLOGICAL SETTING

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

## 1.4 MINERAL RESOURCE ESTIMATE

Mineral Resources for the Project were classified in accordance with NI 43-101 requirements. Table 1.2 summarizes the estimated Measured and Indicated Resource, and Table 1.3 summarizes the estimated Inferred Mineral Resource.

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

**Table 1.2 Spanish Mountain Gold 2012 Measured and Indicated Resource**

| Au Cut-off (g/t) | Tonnes > Cut-off (t) | Grade > Cut-off | | Contained Metal | |
|---|---|---|---|---|---|
| | | Au (g/t) | Ag (g/t) | Au (oz) | Ag (oz) |
| 0.10 | 352,290,000 | 0.34 | 0.65 | 3,820,000 | 7,360,000 |
| 0.15 | 274,400,000 | 0.40 | 0.67 | 3,500,000 | 5,910,000 |
| **0.20** | **216,220,000** | **0.46** | **0.68** | **3,180,000** | **4,730,000** |
| 0.25 | 169,770,000 | 0.52 | 0.68 | 2,850,000 | 3,710,000 |
| 0.30 | 134,470,000 | 0.59 | 0.68 | 2,540,000 | 2,940,000 |
| 0.40 | 87,160,000 | 0.72 | 0.68 | 2,010,000 | 1,910,000 |
| 0.50 | 58,990,000 | 0.85 | 0.69 | 1,610,000 | 1,310,000 |
| 0.60 | 41,370,000 | 0.98 | 0.70 | 1,300,000 | 930,000 |
| 0.70 | 29,970,000 | 1.10 | 0.71 | 1,060,000 | 680,000 |
| 0.80 | 22,360,000 | 1.23 | 0.72 | 880,000 | 520,000 |
| 0.90 | 16,870,000 | 1.35 | 0.72 | 730,000 | 390,000 |
| 1.00 | 12,900,000 | 1.47 | 0.73 | 610,000 | 300,000 |

Note: Tonnages and contained metals may not exactly equal individual tables due to rounding.

**Table 1.3 Spanish Mountain Gold 2012 Inferred Resource**

| Au Cut-off (g/t) | Tonnes > Cut-off (t) | Grade > Cut-off | | Contained Metal | |
|---|---|---|---|---|---|
| | | Au (g/t) | Ag (g/t) | Au (oz) | Ag (oz) |
| 0.10 | 697,310,000 | 0.24 | 0.60 | 5,380,000 | 13,450,000 |
| 0.15 | 459,790,000 | 0.30 | 0.63 | 4,430,000 | 9,310,000 |
| **0.20** | **316,740,000** | **0.36** | **0.65** | **3,650,000** | **6,620,000** |
| 0.25 | 214,940,000 | 0.42 | 0.66 | 2,910,000 | 4,560,000 |
| 0.30 | 147,830,000 | 0.49 | 0.67 | 2,320,000 | 3,180,000 |
| 0.40 | 70,160,000 | 0.65 | 0.70 | 1,470,000 | 1,580,000 |
| 0.50 | 39,320,000 | 0.81 | 0.68 | 1,030,000 | 860,000 |
| 0.60 | 23,850,000 | 0.99 | 0.67 | 760,000 | 510,000 |
| 0.70 | 15,990,000 | 1.15 | 0.67 | 590,000 | 340,000 |
| 0.80 | 11,650,000 | 1.30 | 0.67 | 490,000 | 250,000 |
| 0.90 | 8,620,000 | 1.47 | 0.66 | 410,000 | 180,000 |
| 1.00 | 6,820,000 | 1.60 | 0.63 | 350,000 | 140,000 |

Note: Tonnages and contained metals may not exactly equal individual tables due to rounding.

## 1.5 Mineral Processing and Metallurgical Testing

### *1.5.1 Mineral Processing*

The Project will include a 40,000 t/d processing facility. The plant feed material will follow a conventional comminution process with a primary crushing stage, followed by a semi-autogenous mill in closed circuit with a pebble crusher and by ball mill grinding in closed circuit with cyclone classification. The ball mill/cyclone circuit will include centrifugal gravity concentrators for the recovery of liberated gold particles. The gravity concentrate will be treated in a dedicated intensive cyanidation and electrowinning circuit in order to recover the gold from the gravity concentrate.

The classification cyclone overflow will be the feed to the flotation circuit which will incorporate a pre-flotation stage, as well as a rougher and two open-circuit cleaner stages for upgrading the sulphide-mineral rich gold-bearing flotation concentrate product. A scavenging gravity concentration circuit will treat the pre-flotation concentrate and the cleaner flotation tailings streams. The concentrate produced from this secondary gravity circuit will be combined with the flotation concentrate, while the tailings will be discharged to the tailings storage facility. The tailings from the flotation circuit will also be discharged to the tailings storage facility (TSF).

The flotation concentrate will be thickened and reground to a finer product size in order to improve the leach kinetics of the gold in the subsequent carbon-in-leach (CIL) circuit. Ahead of the CIL, a pre-aeration stage will allow for pH control and the oxidation of any labile sulphide mineral particles. The CIL adsorption circuit will be conventional with carbon moved counterflow to the pulp stream. Loaded carbon will be transferred from the head tank in the CIL circuit for acid washing and subsequent gold elution and electrowinning. The gold produced from the electrowinning circuit will periodically be collected and dried, and subsequently smelted into dore bars. The eluted carbon will be reactivated in a kiln to remove impurities and then screened to remove carbon fines. The regenerated and screened carbon will then be returned to the CIL adsorption circuit.

The slurry exiting the adsorption circuit will pass over a carbon safety screen which will retain any misplaced carbon particles. The screen underflow slurry will be discharged to the detoxification circuit where the cyanide concentration will be reduced to acceptable environmental concentrations prior to disposal of this slurry to the TSF.

Process water will be recycled from the concentrate thickener overflow and will be supplemented by process water recovered from the TSF. Fresh water will be required for gland seal service for the slurry pumps and for reagent make-up. A proportion of the process water will be treated in a water treatment facility for specific use in the gravity concentration circuits.

Reagents will be made up and prepared for use on a regular basis in the plant. Compressed and blower air requirements will be provided as required.

### 1.5.2 METALLURGICAL TESTING

Sample material representative of the two major rock types present in the deposit was collected from drillholes and used in a series of metallurgical test programs. These test programs included comminution, gravity concentration, cyanidation leach work, flotation and flotation concentrate regrind and cyanidation tests, as well as carbon adsorption data, while best-practice reagent dosages have been used for the detoxification of the CIL tailings stream prior to disposal.

The gold was found to occur as fine-grained particles generally less than 5 µm in size. The gold was found to be predominantly associated with quartz and sulphide minerals, mainly pyrite.

The presence of a preg-robbing carbonaceous material in the plant feed material resulted in the test work focussing on gold recovery ahead of the leaching circuit using gravity concentration, followed by flotation of the gold into a sulphide-rich concentrate which was subsequently reground and leached in a CIL gold recovery circuit. To counter the effect of the carbonaceous material, the flotation test work incorporated a pre-flotation circuit with the resulting high-carbon concentrate disposed of as tailings. The balance of the flotation circuit used a conventional collector reagent to produce the concentrate for the subsequent gold recovery process in the CIL circuit. Test work indicated that the reagent carboxymethyl cellulose successfully depresses the remaining carbonaceous material in the flotation circuit. A secondary gravity concentration circuit was able to scavenge a significant proportion of the gold present in the flotation cleaner tailings streams and the pre-flotation concentrate prior to disposal. The test work programs also defined the crushing and grinding parameters required for the design of the processing plant facilities.

The test work indicated that the overall gold recovery that will be attained in the first three years of production will be 90.3%, with 21.6% coming from the gravity circuit and the balance coming from the CIL circuit. The overall gold recovery will drop to 87.0% in the subsequent years of production in line with the reduced gold plant feed grade value. The overall silver recovery will remain constant at 25.0%.

## 1.6 MINING

The Spanish Mountain deposit will be mined using a conventional open pit mining method, using off-highway haul trucks and hydraulic shovels. The waste and mineralized material will be drilled and blasted, using typical grade control methods and blast-hole sampling.

The open pit was designed for an approximately 14-year LOM. The potential in-pit resource, based on a 0.20 g/t gold cut-off, is summarized in Table 1.4.

**Table 1.4 Potential In-pit Resource Estimate**

| | Unit | Amount |
|---|---|---|
| Measured and Indicated Resource Class | kt | 167,198 |
| Gold Grade | g/t | 0.477 |
| Silver Grade | g/t | 0.675 |
| Inferred Resource Class | kt | 38,700 |
| Gold Grade | g/t | 0.498 |
| Silver Grade | g/t | 0.667 |
| Total All Classes | kt | 205,898 |
| Gold Grade | g/t | 0.481 |
| Silver Grade | g/t | 0.673 |
| Waste Material | kt | 464,874 |
| Strip Ratio | t/t | 2.3 |

## 1.7 PROJECT INFRASTRUCTURE

The Property is located in the Cariboo region of central BC, 6 km east of the community of Likely, and 66 km northeast of the City of Williams Lake.

The Property can be reached from Williams Lake via the Likely road, which is a paved secondary road that leaves Highway 97 at 150 Mile House, approximately 16 km south of Williams Lake, and continues for 87 km to Likely. From Likely, the

Property is accessed from the Spanish Mountain 1300 FSR. This road currently travels through the proposed mine site; it will require rerouting in order to accommodate the location of the north waste dump and open pit. Access to this FSR route through the site will be maintained throughout the LOM.

The Project will require the construction of a number of infrastructure items, including:

- access and haul roads
- a primary crushing building
- a coarse crush stockpile
- a pebble crushing building
- a mill building
- an administration and mine dry building
- a maintenance and truck shop building
- an assay laboratory
- a cold storage warehouse.

Figure 1.2 illustrates the overall Project site layout.

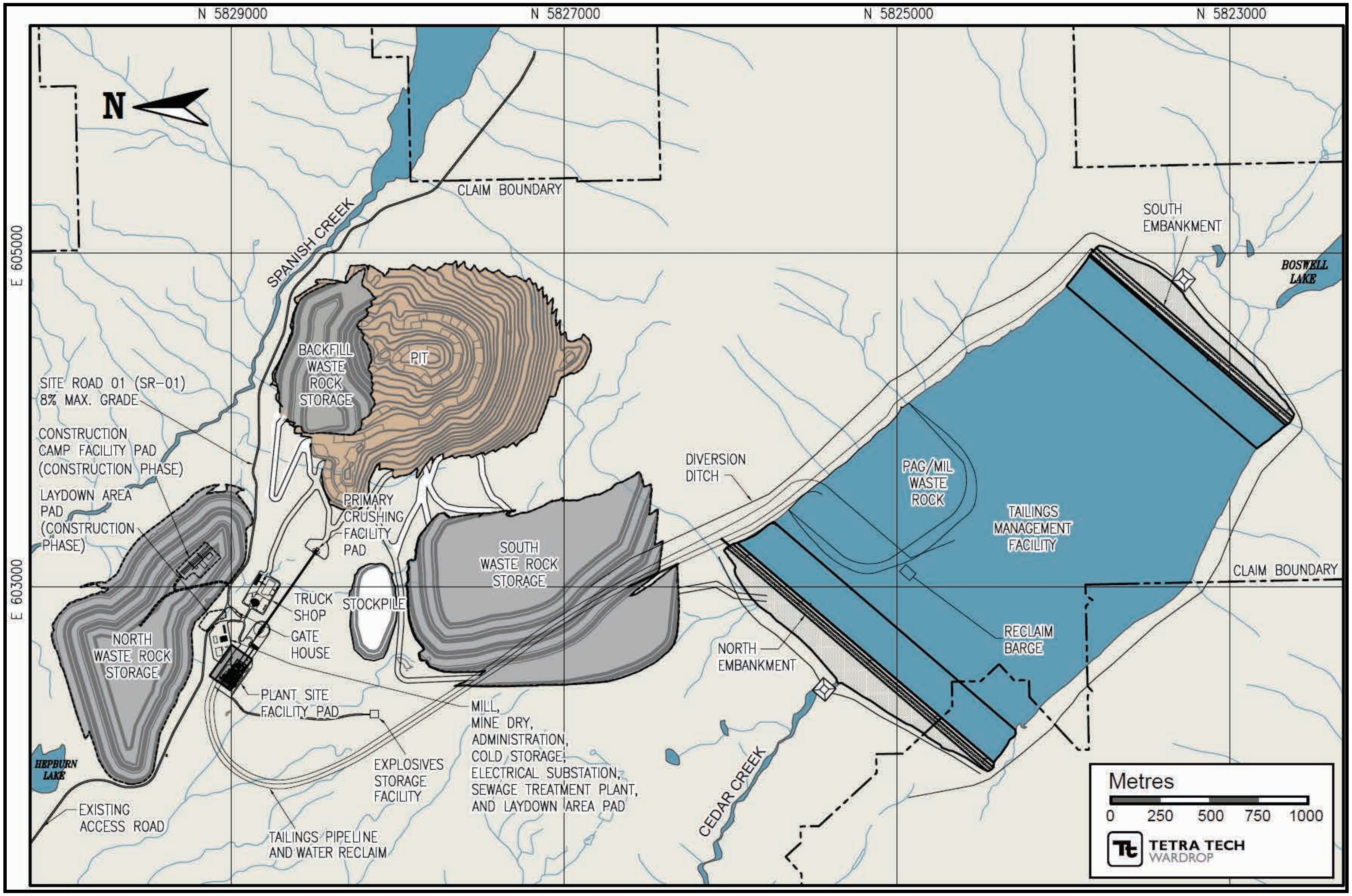
All buildings and facilities will be constructed with appropriate heating, ventilation and air conditioning (HVAC) and fire protection systems, water and plumbing systems, and fire protection and dust control systems.

A series of mine haul roads will be constructed from the open pit to the primary crusher, and as well as site roads to and from the truck shop, tailings storage areas, and WRSAs.

TETRA TECH

Spanish Mountain Gold Ltd.

**Figure 1.2 General Arrangement Layout**



### 1.7.1 *Power Supply to Plant Site*

The Project requires 60 MW of peak load for 40,000 t/d operation demand. A new transmission line interconnecting the SMG site to BC Hydro's power system is required to meet power requirement in operation. Stantec evaluated six power supply options, including preliminary design basis, cost estimate, bill of materials and development schedule.

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

### 1.7.2 *Waste and Water Management*

The principal objective of the TSF is to provide secure containment of all tailings solids and potentially acid generating (PAG)/metal leaching (ML) waste.

The processing plant will produce two tailings streams: rougher tailings and cleaner tailings, which will be transported from the plant site to the TSF in separate pipelines at an average solids content of 38% by weight for the rougher tailings and 43% by weight for the cleaner tailings. Each tailings stream will be deposited independently; the rougher tailings will be discharged along the TSF embankments to create tailings beaches and the cleaner tailings, which are assumed to be PAG and ML, will be discharged subaqueously to allow for progressive encapsulation by the rougher tailings and saturation by the supernatant pond. The PAG/ML waste will also be deposited to allow for progressive encapsulation by the rougher tailings and saturation by the supernatant pond.

The TSF starter embankment will be constructed during the pre-production phase and is sized to store the estimated volume of tailings and PAG/ML waste produced during the first two years of operation, plus the supernatant pond volume with allowances for wave run-up, post-seismic settlement, sloping beaches and containment of the inflow design flood. The TSF embankments will be constructed in stages with each stage providing the required capacity for the period until the next stage is complete. The final configuration allows for storage of approximately 205 Mt of tailings, 35 Mt of PAG/ML waste, plus the supernatant pond volume and freeboard allowances.

## 1.8 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.

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. Wildlife species are typical for the region.

The Project will require approval under the federal and provincial environmental assessment (EA) process prior to receiving the necessary permits and authorizations for construction and operation. The federal *Fisheries Act* prohibits the harmful alteration, disruption, or destruction of fish habitat without specific authorization. Construction of the TSF in the Nina Lake basin of the Cedar Creek watershed may require a Schedule 2 Amendment under the Metal Mining Effluent Regulations (MMER) of the *Fisheries Act*. Fish habitat compensation will be required to balance the loss of 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 Environmental Assessment Office (EAO) Project Information Centre (e-PIC) and federal Canadian Environmental Assessment Agency (CEAA) registry.

The Project is situated within the asserted traditional territories of the T'exelc (Williams Lake) and Xats'ull/Cmetem' (Soda Creek) First Nations, as well as the Lhtako Dene Nation (Red Bluff Indian Band). SMG has signed cooperation agreements with each of the three First Nations. These agreements govern the participation of each party during the EA and permitting review of the Project.

# 1.9 Capital and Operating Costs

## 1.9.1 Capital Cost Estimate

The total estimated pre-production capital cost for the design, construction, and installation and commissioning for all facilities and equipment is CAD$763.1 million. A breakdown of the total is shown in Table 1.5.

This estimate has been prepared in accordance with the Class 4 Prefeasiblity Cost Estimate standards of the Association for the Advancement of Cost Engineering International (AACE). The accuracy of the estimate is ±35% unless otherwise noted.

This study has been prepared with a base date of Q4 2012 with no provision for escalation.

**Table 1.5 Capital Cost Summary**

| Description | Capital Cost CAD$ million |
|---|---|
| **Direct Costs** | |
| Overall Site | 20.1 |
| Open Pit Mining | 128.9 |
| Mineralized Material Handling | 54.8 |
| Process | 169.7 |
| Tailings and Water Management | 70.4 |
| Environmental | 12.0 |
| On-site Infrastructure | 57.0 |
| Off-site Infrastrucutre | 16.3 |
| **Subtotal** | **529.3** |
| Indirect Costs | 130.2 |
| Owner's Costs | 16.7 |
| Contingencies | 86.9 |
| **Total** | **763.1** |

## 1.9.2 Operating Cost Estimate

On site operating costs are estimated to be CAD$10.78/t of material milled including mining, processing, general and administrative (G&A), and plant services. The unit costs summarized in Table 1.6 are based on an annual production rate of 40,000 t/d, and 365 d/a of operation.

**Table 1.6 Operating Cost Summary**

| Area | Unit Cost (CAD$/t milled) |
|---|---|
| Mining | 5.24 |
| Processing | 4.49 |
| Tailings | 0.04 |
| G&A | 0.59 |
| Off-site Costs (Including Royalty) | 0.42 |
| **Total Operating Cost** | **10.78** |

## 1.10 ECONOMIC ANALYSIS

An economic evaluation of the Project was carried out in US dollars by Tetra Tech incorporating all the relevant capital, operating, working, sustaining costs, and royalties (1.5% of NSR). The evaluation was based on a pre-tax financial model. For the 15-year mine life and 206 Mt resource inventory, the following pre-tax financial parameters were calculated using the base case gold price:

- 15% IRR
- 4.4-year payback on US$756 million capital
- US$454 million NPV at 5% discount value.

SMG commissioned PricewaterhouseCoopers LLP (PwC) in Vancouver, BC to prepare a tax model for the post-tax economic evaluation of the Project with the inclusion of applicable income and mining taxes. (see Section 22.0).

The following post-tax financial parameters were calculated:

- 12% IRR
- 4.7-year payback on US$756 million capital
- US$291 million NPV at an 5% discount rate.

The gold and silver prices used for the base case are US$1,462/oz and US$28.13/oz respectively, using the three-year trailing average (as of November 1, 2012). The base case exchange rate was set at US$0.9905:CAD$1.00, also using the three-year trailing average.

Sensitivity analyses were developed to evaluate the Project economics.

## 1.11 PROJECT SCHEDULE

A high-level schedule has been prepared for the Project and is provided in Section 24.0. The schedule has been developed with the goal of developing a sustainable project that includes adherence to global mining practice professional standards; providing long-lasting benefits to local communities, a high class of project management that results in environmental protection, health and safety.

## 1.12 OPPORTUNITIES AND RECOMMENDATIONS

Based on the work carried out in this PEA and the resultant economic evaluation, this study should be followed by further technical and economic studies leading to a feasibility study in order to further assess the economic viability of the Project.

Detailed opportunities and recommendations are provided in Section 26.0 of this report.

# 2.0 INTRODUCTION

## 2.1 TERMS OF REFERENCES AND PURPOSE OF REPORT

SMG commissioned Tetra Tech to complete a technical report and PEA on the Project, in accordance with Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Best Practices, and with NI 43-101 Standards of Disclosure for Mineral Projects.

A summary of the Qualified Persons (QPs) responsible for this report is provided in Table 2.1. The following QPs conducted site visits of the Property:

- Gary Giroux, M.A.Sc., P.Eng. conducted a site visit on June 29, 2011.
- Agnes Koffyberg, M.Sc., P.Geo. conducted a site visit on July 10, 2012.
- Marinus Andre De Ruijter, P.Eng., conducted a site visit on September 27, 2011.
- Les Galbraith, P.Eng. conducted a site visit on October 4, 2012.
- Ibro Hadzismajlovic, P.Eng. conducted site visits from July to November 9, 2011.

**Table 2.1 Summary of QPs**

| Report Section | | Company | QP |
|---|---|---|---|
| **1.0** | Summary | All | Sign-off by Section |
| **2.0** | Introduction | Tetra Tech | Andrea Cade, M.Sc., P.Geo. |
| **3.0** | Reliance on Other Experts | Tetra Tech | Andrea Cade, M.Sc., P.Geo. |
| **4.0** | Property Description and Location | Discovery | Agnes Koffyberg, M.Sc., P.Geo. |
| **5.0** | Accessibility, Climate, Local Resources, Infrastructure, and Physiography | Discovery | Agnes Koffyberg, M.Sc., P.Geo. |
| **6.0** | History | Discovery | Agnes Koffyberg, M.Sc., P.Geo. |
| **7.0** | Geological Setting and Mineralization | Discovery | Agnes Koffyberg, M.Sc., P.Geo. |
| **8.0** | Deposit Types | Discovery | Agnes Koffyberg, M.Sc., P.Geo. |
| **9.0** | Exploration | Discovery | Agnes Koffyberg, M.Sc., P.Geo. |
| **10.0** | Drilling | Discovery | Agnes Koffyberg, M.Sc., P.Geo. |
| **11.0** | Sample Preparation, Analyses, and Security | Discovery | Agnes Koffyberg, M.Sc., P.Geo. |
| **12.0** | Data Verification | Discovery | Agnes Koffyberg, M.Sc., P.Geo. |
| **13.0** | Mineral Processing and Metallurgical Testing | Tetra Tech | Marinus Andre De Ruijter, P.Eng. |
| **14.0** | Mineral Resource Estimates | Giroux Consultants | Gary Giroux, M.A.Sc., P.Eng. |
| **15.0** | Mineral Reserve Estimates | MMTS | Bob Fong, P.Eng. |

*table continues…*

| Report Section | | Company | QP |
|---|---|---|---|
| **16.0** | Mining Methods | MMTS<br>BGC | Bob Fong, P.Eng.<br>H. Warren Newcomen, P.Eng. |
| **17.0** | Recovery Methods | Tetra Tech | Marinus Andre De Ruijter, P.Eng. |
| **18.0** | Infrastructure | - | - |
| | **18.1** Overview | Tetra Tech | Hassan Ghaffari, P.Eng. |
| | **18.2** Site Layout | Tetra Tech | Hassan Ghaffari, P.Eng. |
| | **18.3** Roads | Tetra Tech | Hassan Ghaffari, P.Eng. |
| | **18.4** Buildings | Tetra Tech | Hassan Ghaffari, P.Eng. |
| | **18.5** Building Services | Tetra Tech | Hassan Ghaffari, P.Eng. |
| | **18.6** Water Supply and Distribution | Tetra Tech | Hassan Ghaffari, P.Eng. |
| | **18.7** Waste Disposal | Tetra Tech | Hassan Ghaffari, P.Eng. |
| | **18.8** Fuel Storage | Tetra Tech | Hassan Ghaffari, P.Eng. |
| | **18.9** On-site Explosives Storage | Tetra Tech | Bob Fong, P.Eng. |
| | **18.10** Power Supply to Plant Site | Stantec | Ibro Hadzismajlovic, P.Eng. |
| | **18.11** Tailings Storage Facility | Knight Piésold | Les Galbraith, P.Eng. |
| | **18.12** Waste Rock Management | Knight Piésold | Les Galbraith, P.Eng. |
| | **18.13** Construction Camp Accommodation | Tetra Tech | Hassan Ghaffari, P.Eng. |
| | **18.14** Communications | Tetra Tech | Hassan Ghaffari, P.Eng. |
| **19.0** | Market Studies and Contracts | Tetra Tech | Hassan Ghaffari, P.Eng. |
| **20.0** | Environmental Studies, Permitting, and Social or Community Impact | Knight Piésold | Ken Brouwer, P.Eng. |
| **21.0** | Capital and Operating Costs | - | - |
| | **21.1** Summary | Tetra Tech | Harvey Wayne Stoyko, P.Eng. |
| | **21.2** Capital Cost Estimate | - | - |
| | 21.2.1 Summary of Capital Costs | Tetra Tech | Harvey Wayne Stoyko, P.Eng. |
| | 21.2.2 Estimating Methodology | Tetra Tech | Harvey Wayne Stoyko, P.Eng. |
| | 21.2.3 Responsibility | Tetra Tech | Harvey Wayne Stoyko, P.Eng. |
| | 21.2.4 Direct Costs | Tetra Tech | Harvey Wayne Stoyko, P.Eng. |
| | 21.2.5 Indirect Costs | Tetra Tech | Harvey Wayne Stoyko, P.Eng. |
| | 21.2.6 Owner's Costs | Tetra Tech | Harvey Wayne Stoyko, P.Eng. |
| | 21.2.7 Contingencies | Tetra Tech | Harvey Wayne Stoyko, P.Eng. |
| | 21.2.8 Mining Capital Cost Estimate | MMTS | Bob Fong, P.Eng. |
| | 21.2.9 Waste and Water Management Capital Cost Estimate | Knight Piésold | Les Galbraith, P.Eng. |
| | 21.2.10 Capital Costs Exclusions | Tetra Tech | Harvey Wayne Stoyko, P.Eng. |
| | **21.3** Operating Cost Estimate | - | - |
| | 21.3.1 Summary of Operating Costs | Tetra Tech | Marinus Andre De Ruijter, P.Eng. |
| | 21.3.2 Mining Operating Cost Estimate | MMTS | Bob Fong, P.Eng. |
| | 21.3.3 Process Operating Cost Estimate and G&A Costs | Tetra Tech | Marinus Andre De Ruijter, P.Eng. |
| | 21.3.4 Waste and Water Management Operating Costs | Knight Piésold | Les Galbraith, P.Eng. |

*table continues…*

| Report Section | | Company | QP |
|---|---|---|---|
| | 21.3.5 Fisheries Compensation and Environmental Monitoring | Knight Piésold | Ken Brouwer, P.Eng. |
| **22.0** | Economic Analysis | Tetra Tech | Sabry Abdel Hafez, Ph.D., P.Eng. |
| **23.0** | Adjacent Properties | Discovery | Agnes Koffyberg, M.Sc., P.Geo. |
| **24.0** | Other Relevant Data and Information | Discovery/ Tetra Tech | Agnes Koffyberg, M.Sc., P.Geo./ Hassan Ghaffari, P.Eng. |
| **25.0** | Interpretation and Conclusions | All | Sign-off by Section |
| **26.0** | Recommendations | All | Sign-off by Section |
| **27.0** | References | All | Sign-off by Section |

### 2.1.1 UNITS OF MEASUREMENT

All units of measurement used in this technical report are in metric.

## 2.2 INFORMATION AND DATA SOURCES

In the preparation of this report, various historical engineering, geological and management reports about the Project or site were reviewed and supplemented by direct site examinations and investigations. All data files reviewed for this study were provided by SMG in the form of hard copy documents, Adobe Acrobat® files, Microsoft Excel® files, email correspondence, and personal communications with management and personnel from SMG. Work completed by SMG includes drilling and sampling, trenching, metallurgical testing, and geophysical surveying.

As the author of the resource estimate in Section 14.0, Mr. Gary Giroux, M.A.Sc., P.Eng. updated his previous resource estimate, based on additional data generated by SMG in late 2011 and in 2012. A previous resource estimate was presented by Mr. Giroux and Ms. Koffyberg in a 2011 report entitled "Updated Resource Estimation Report on the Spanish Mountain Gold Deposit", dated November 30, 2011 and available on SEDAR.

In the preparation of the geological information, Ms. Koffyberg used a variety of unpublished company data, as well as corporate news releases, geological reports, geological maps and mineral claim maps, sourced from both provincial and federal governments. The principal sources of geological information have been the reports by Page (2003), Singh (2008), Peatfield et al. (2009), as well as assessment reports and some scientific papers, including Rhys et al. (2009). Valuable site-specific information was provided by Brian Groves, President and CEO of SMG and by Judy Stoeterau, Vice President, Geology for SMG. Geology figures have been prepared by SMG geologists.

A complete list of references is provided in Section 27.0.

# 3.0 RELIANCE ON OTHER EXPERTS

In preparation of this report, Tetra Tech has relied upon others for information, and disclaims responsibility for information derived from reports pertaining to mineral tenure, property ownership, surface rights, environment, royalties, and social issues. Neither Tetra Tech nor the authors are qualified to provide extensive comment on legal issues, including mineral tenure status associated with the Project, and ownership is provided for general purposes only. Research into the mineral claim status was limited to the information available on British Columbia MTO website.

Tetra Tech has reviewed and analyzed data and reports provided by SMG, together with publicly available information, drawing its own conclusions augmented by direct field examination. Information from third party sources is quoted or referenced throughout this technical report and is fully disclosed in Section 27.0. Tetra Tech used information from these sources under the assumption that the contents are accurate.

Assay work applicable to the resource estimate was done by ALS Minerals (ALS) of North Vancouver, BC, and to a lesser extent by Eco-tech Laboratories Ltd (Eco-tech) of Kamloops, BC; and by Acme Analytical Laboratories Ltd. of Vancouver, BC. All are International Organization for Standardization (ISO) certified labs. Diamond drilling was carried out by Atlas Drilling Company of Kamloops, BC from 2010 to 2012; by LDS Diamond Drilling Ltd. of Kamloops, BC in 2005 through 2009; while North Star Drilling Ltd. of Delta, BC also carried out some drilling in 2007 to 2008. Recent 2011 and 2012 surveying was done in-house using Trimble R8R2K Survey GPS equipment supplied by Cansel Survey Equipment Inc. Previous survey work was performed by Crowfoot Developments Ltd. of Kamloops and by Allnorth Consultants of Prince George.

The QPs who prepared this report relied on information provided by the following experts who are not QPs:

- Mr. R.C. Brodie, R.P.Bio, of Knight Piésold, for matters relating to environmental studies, permitting, and social or community impact in Section 20.0.
- Mr. Tim Johnston of PwC for matters relating to a tax model for the post-tax economic evaluation in Section 22.0.

# 4.0 PROPERTY DESCRIPTION AND LOCATION

The following section is derived directly from the report “Technical Report on an Updated Mineral Resource Estimate on the Spanish Mountain Gold Deposit”, by G. Giroux and A. Koffyberg and dated August 31, 2012. Minor changes have been made for report consistency.

## 4.1 LOCATION

The Property is located in south-central BC, approximately 6 km southeast of the community of Likely and 66 km northeast of the City of Williams Lake (Figure 4.1). The Property is situated between Quesnel Lake and Spanish Lake with the centre at approximately latitude 52˚ 34' north and longitude 121˚ 28' west. The main resource, consisting of the Main and North Zones, is located west of the northwest end of Spanish Lake, and is centred at approximate UTM coordinates 604425 East and 5827900 North (Datum NAD83, Zone 10). It is located mainly within the mineral claim 204667 as well as mineral claims 204225 and 204226.

**Figure 4.1 Property Location**

## 4.2 DESCRIPTION

The Property consists of 46 MTO mineral claims, of which 20 are legacy claims. Of the 46 claims, 2 lie on the west side of Quesnel Lake; the other 44 form a contiguous block of claims covering an area of approximately 7,680 ha (Figure 4.2). The mineral claims lie on BC Mineral TRIM Map Sheets 093A.053, 054 and 063. All claims are 100% owned by SMG. Table 4.1 lists the details of the claim tenures. SMG also owns 13 overlying placer claims in the area (Figure 4.3). The Property contains surface rights of several private home owners along the eastern side of Quesnel Lake and one, small isolated parcel (DL12083) at the northwest end of Spanish Lake (Figure 4.4). In addition, a third party(s) owns six placer leases (Figure 4.4).

## 4.3 OWNERSHIP

SMG owns all 46 mineral claims comprising the Property. The company was formerly named Skygold Ventures Ltd. (Skygold), with the change in name effective January 14, 2010. Information on four underlying option agreements pertaining to the tenure status has been provided by SMG and pertains to certain claims, as highlighted in Table 4.1. These agreements are summarized as follows:

- a 2.5% NSR royalty payable to Robert E. Mickle (Mickle)
- a 2.5% NSR royalty payable to D.E. Wallster (Wallster) and J.P. McMillan (McMillan)
- a 2.5% NSR royalty payable to G. Richmond (Richmond) on the two Cedar Creek claims
- a 4% NSR royalty payable to Acrex Ventures Ltd on the 10 Acrex claims.

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

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

**Figure 4.2 Mineral Claim Location**

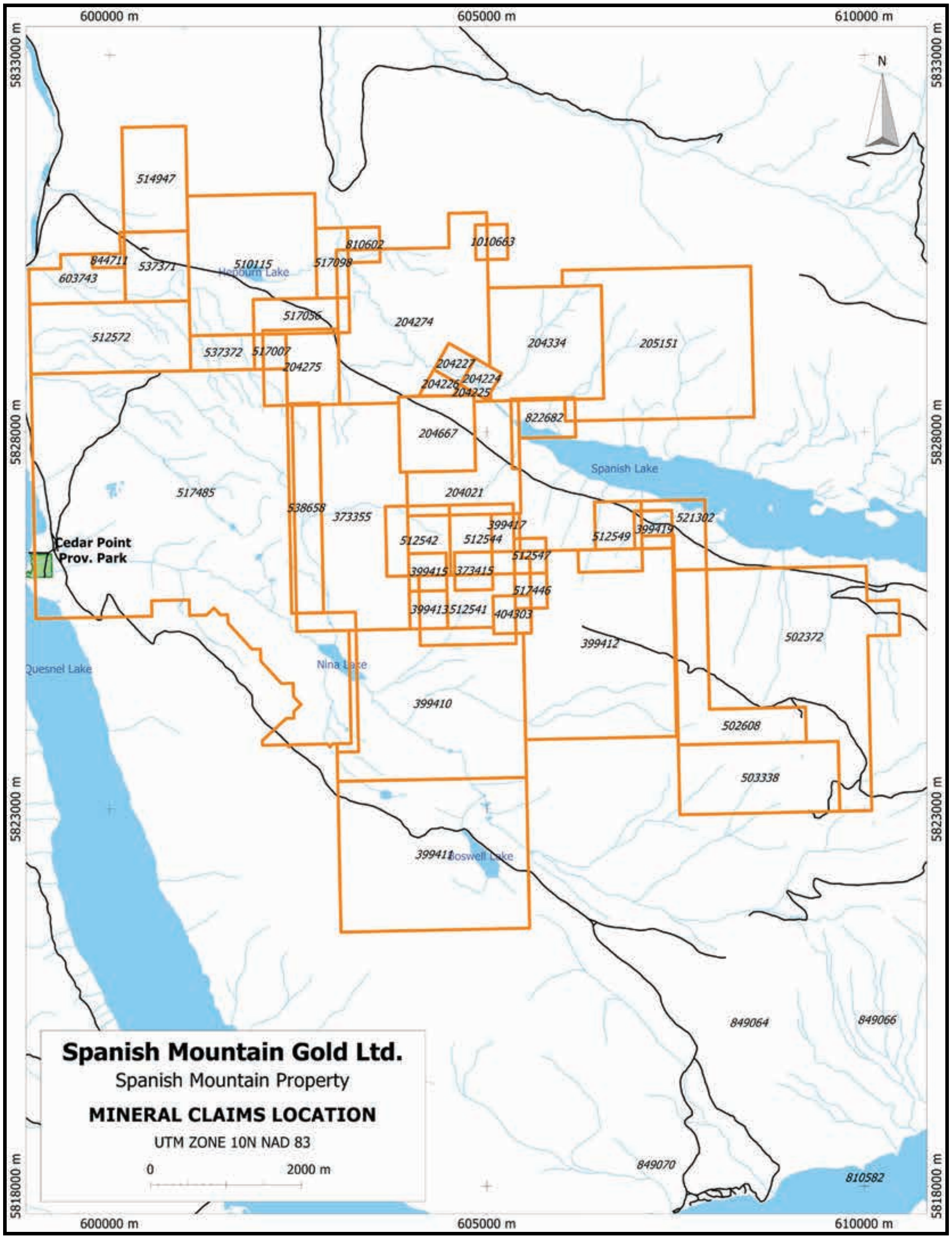


**Figure 4.3 Placer Claim Location**

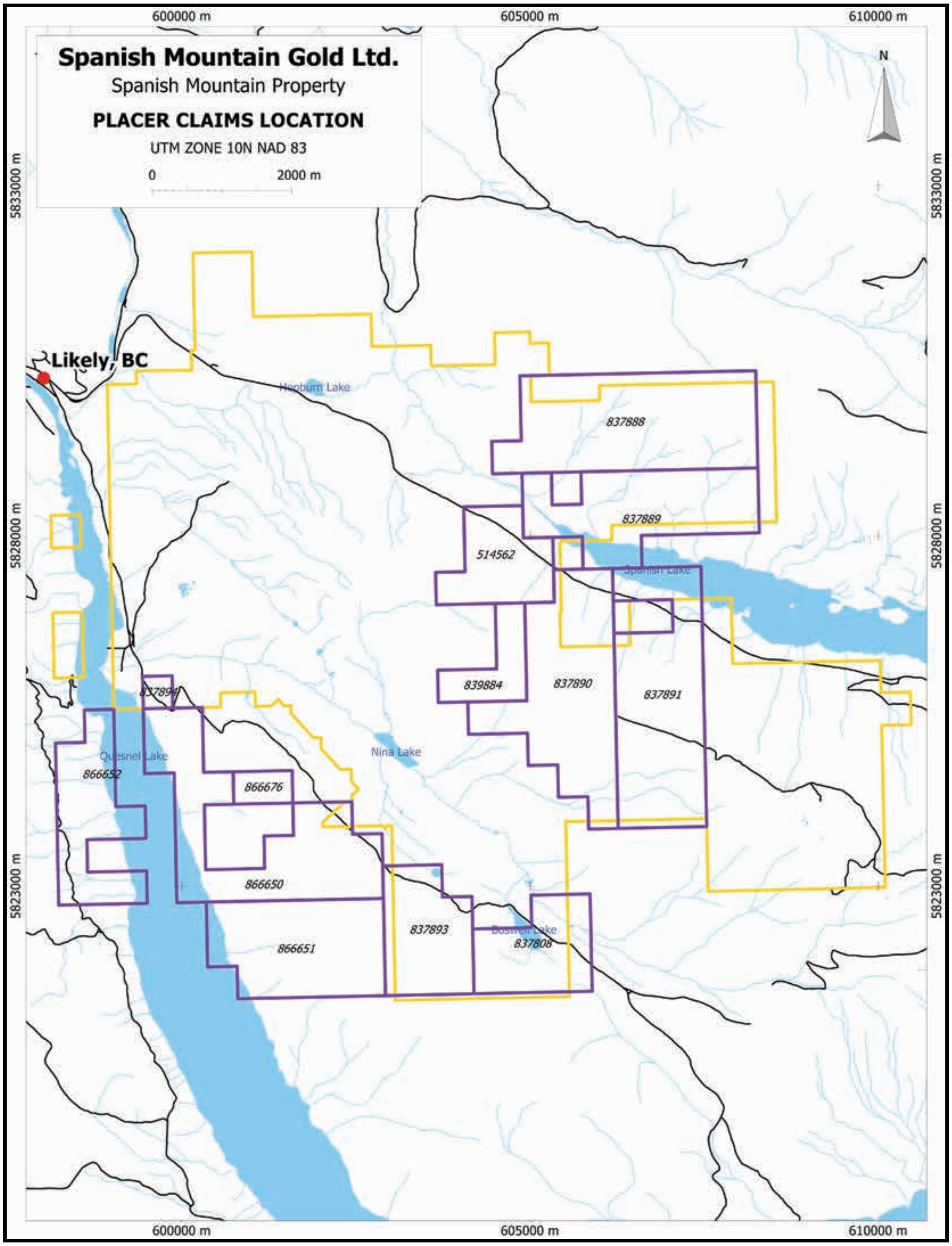


**Figure 4.4 Surface Rights Location**

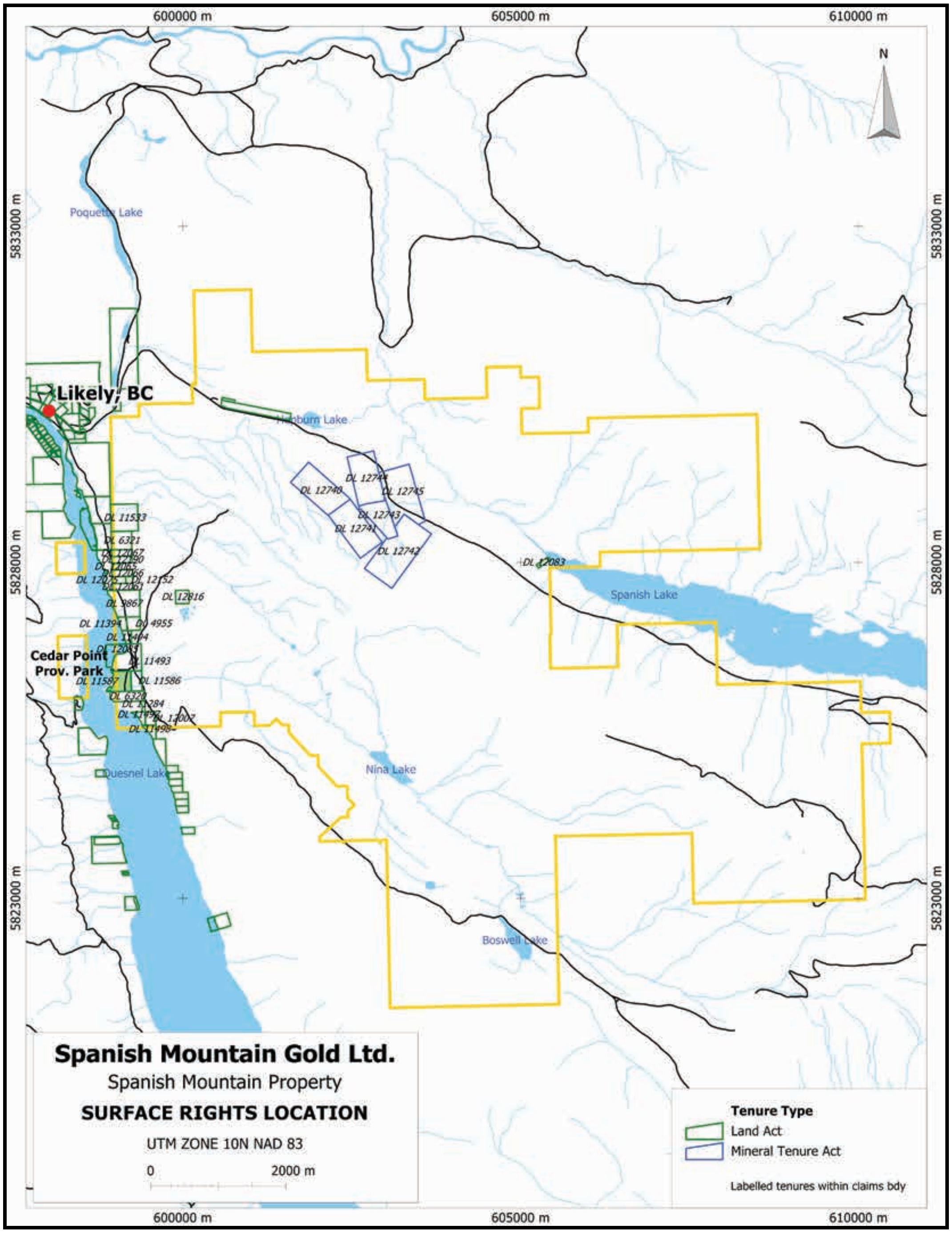


**Table 4.1 Mineral Tenure Description**

| Tenure Number | Claim Name | Area (ha) | Map Number | Registered Owner | Expiry Date |
|---|---|---|---|---|---|
| 204021 | PESO | 225.00 | 093A.053 | SMG | 2021/Nov/01 |
| 204224 | DON 1 | 25.00 | 093A.053 | SMG | 2021/Nov/01 |
| 204225 | DON 2 | 25.00 | 093A.053 | SMG | 2021/Nov/01 |
| 204226 | DON 3 | 25.00 | 093A.053 | SMG | 2021/Nov/01 |
| 204227 | DON 4 | 25.00 | 093A.053/063 | SMG | 2021/Nov/01 |
| 204274 | MARCH 1 | 500.00 | 093A.053/063 | SMG | 2021/Nov/01 |
| 204275 | MARCH 2 | 100.00 | 093A.053/063 | SMG | 2021/Nov/01 |
| 204334 | JUL 2 | 225.00 | 093A.053/063 | SMG | 2021/Nov/01 |
| 204667 | CPW | 100.00 | 093A.053 | SMG | 2021/Nov/01 |
| 205151 | MEY 1 | 500.00 | 093A.053/063 | SMG | 2021/Nov/01 |
| 373355 | ARMADA | 450.00 | 093A.053 | SMG | 2021/Nov/01 |
| 373415 | N.R.1 | 25.00 | 093A.053 | SMG | 2021/Nov/01 |
| 399410 | ARMADA 2 | 500.00 | 093A.053 | SMG | 2021/Nov/01 |
| 399411 | ARMADA 4 | 500.00 | 093A.053 | SMG | 2021/Nov/01 |
| 399412 | ARMADA 5 | 500.00 | 093A.053 | SMG | 2021/Nov/01 |
| 399413 | ARMADA 6 | 25.00 | 093A.053 | SMG | 2021/Nov/01 |
| 399415 | ARMADA 8 | 25.00 | 093A.053 | SMG | 2021/Nov/01 |
| 399417 | ARMADA 10 | 25.00 | 093A.053 | SMG | 2021/Nov/01 |
| 399419 | ARMADA 12 | 25.00 | 093A.053 | SMG | 2021/Nov/01 |
| 403303 | AG 2 | 25.00 | 093A.053 | SMG | 2021/Nov/01 |
| 512541 | - | 117.89 | 093A.053 | SMG | 2021/Nov/01 |
| 502372 | SPANISH 1 | 491.33 | 093A.053/054 | SMG | 2015/Nov/08 |
| 502608 | SPANISH 2 | 157.23 | 093A.053/054 | SMG | 2015/Nov/08 |
| 503338 | SPANISH 3 | 196.58 | 093A.053/054 | SMG | 2015/Nov/08 |
| 510115 | GOLDEN AIRPORT | 274.82 | 093A.063 | SMG | 2020/Dec/12 |
| 512542 | - | 78.58 | 093A.053 | SMG | 2021/Nov/01 |
| 512544 | - | 78.58 | 093A.053 | SMG | 2021/Nov/01 |
| 512547 | - | 19.65 | 093A.053 | SMG | 2021/Nov/01 |
| 512549 | - | 78.58 | 093A.053 | SMG | 2021/Nov/01 |
| 512572 | FISCHER CREEK | 196.34 | 093A.063 | SMG | 2013/May/01 |
| 514947 | GOLD TREND | 117.76 | 093A.063 | SMG | 2020/Dec/12 |
| 517007 | GOLD | 19.64 | 093A.063 | SMG | 2020/Dec/12 |
| 517056 | GOLDIE | 58.90 | 093A.063 | SMG | 2020/Dec/12 |
| 517098 | GOLD3 | 39.26 | 093A.063 | SMG | 2020/Dec/12 |
| 517446 | - | 19.65 | 093A.053 | SMG | 2021/Nov/01 |
| 517485 | - | 1335.78 | 093A.053 | SMG | 2021/Jul/28 |
| 521302 | AKV | 58.94 | 093A.053 | SMG | 2015/Nov/08 |
| 537371 | MOOREHEAD 12 | 78.52 | 093A.063 | SMG | 2020/Dec/12 |
| 537372 | MOOREHEAD 13 | 39.27 | 093A.063 | SMG | 2020/Dec/12 |
| 538658 | MOREHEAD 14 | 117.86 | 093A.053 | SMG | 2021/Aug/04 |

*table continues…*

| Tenure Number | Claim Name | Area (ha) | Map Number | Registered Owner | Expiry Date |
|---|---|---|---|---|---|
| 603743 | LIKELY GULCH | 78.52 | 093A.063 | SMG | 2013/Nov/01 |
| 748902 | SPAN 1 | 39.29 | 093A.053 | SMG | 2013/Nov/01 |
| 810602 | SPAN 3 | 19.63 | 093A.063 | SMG | 2013/May/01 |
| 822682* | - | 78.56 | 093A.053 | SMG | 2021/Nov/01 |
| 844711 | SPAN 4 | 19.63 | 093A.063 | SMG | 2013/May/01 |
| 851442 | SPAN 9 | 19.64 | 093A.053 | SMG | 2013/Nov/01 |
| **Total** | **-** | **7,680.43** | **-** | **-** | **-** |

Notes: Claims in red are subject to the Mickle option agreement.
Claim in blue is subject to the Wallster and McMillan option agreement.
Claims in green are subject to the Cedar Creek purchase agreement.
Claims in purple are subject to the Acrex purchase agreement.
*Claim 822682 is converted from legacy claim 204727 (MY 1), which is subject to the Mickle option agreement.

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

An option agreement dated January 20, 2003 between Wildrose (the Optionee), SMG (the Assignee), and 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 $348,000 over nine years, in addition to 30,000 common shares of the Assignee on signing. All of these obligations have been met. There is a 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 $500,000 to the Underlyers at the commencement of commercial production from the CPW claim.

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

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

Under the proposed Transaction, Wildrose shareholders would receive 0.82 common shares of SMG for each common share of Wildrose. SMG would assume outstanding warrants and stock options of Wildrose on the basis that each warrant or option of Wildrose 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 July 9, 2008, SMG announced that "… all the conditions to the acquisition by Spanish Mountain Gold of Wildrose Resources Ltd pursuant to a plan of arrangement under the Business Corporations Act (British Columbia), have been satisfied and the acquisition has now been completed." By virtue of the merger, SMG became responsible for the underlying agreements. Further to this, by virtue of the name change in 2010, SMG is now responsible for the underlying agreements.

Details of the third underlying agreement on the Cedar Creek claims with Cedar Mountain Exploration Inc (Cedar Mountain) are as follows:

A purchase agreement dated June 15, 2010 between SMG and Cedar Mountain, for SMG to earn a 100% interest in 2 mineral claims. The agreement provided for a cash payment totalling $500,000 on signing. There is provision for a 2.5% NSR royalty payable to G. Richmond for any production from these claims, which may be purchased by SMG through the payment to the holder of $500,000 per 1% to G. Richmond.

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

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

The author is not responsible for such information concerning the legal title or legality of any underlying agreements that exist concerning the Property, but has assumed it to be accurate for the purposes of this report.

## 4.4 Taxes and Assessment Work Requirements

Exploration work has advanced the Property such that the majority of the mineral claims are in good standing up to November 1, 2021, as shown on Table 4.1. Six peripheral mineral claims have an earlier expiry of either May 1, 2013 or November 1, 2013. The newly acquired Acrex claims expire in 2015 and 2020.

In order to maintain a mineral tenure in good standing, either exploration work or payment instead of exploration or development (PIED) to the amount required must be submitted prior to the expiry date. The amount required is specified by Section 8.4 of the British Columbia *Mineral Tenure Act* Regulation. This regulation states that the amount of exploration and development work required to maintain a mineral claim is:

- $5/ha during each of the first and second anniversary years
- $10/ha for the third and fourth anniversary years
- $15/ha for the fifth and sixth anniversary years
- $20/ha for each subsequent anniversary year.

The PIED filing cost is charged at double the rate of the assessment work.

The newly adjusted assessment work requirements came into effect on July 1, 2012, and as an aid in this adjustment, all mineral claims are treated as if they are in their first anniversary year for assessment purposes. That is, for example, all claims in the first two years after the July 1 good-to-date will require $5/ha of exploration costs.

Up to ten years of work or PIED can be applied on a claim. In order to obtain credit for the work done on the Property, SMG must file online a Statement of Work (SOW) Exploration and Development Work/Expiry Date Change, and submit an assessment report documenting the results of the work done on the Property. This report must also include an itemized statement of costs.

There are no taxes applicable to the mineral tenures of the Property.

## 4.5 PERMITS AND LIABILITIES

Reclamation bonds for the Property totalling $133,000 are held in trust by the Government of BC, to cover the cost of reclamation on the Property. Since the Project is ongoing, the bonds remain outstanding. There is also a Free Use Permit, which allows for limited tree removal and is good until December 31, 2012. To the best of the author’s knowledge, there are no outstanding environmental issues that would likely to delay or adversely affect the Project.

# 5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The following section is derived directly from the report "Technical Report on an Updated Mineral Resource Estimate on the Spanish Mountain Gold Deposit", by G. Giroux and A. Koffyberg and dated August 31, 2012. Minor changes have been made for report consistency.

## 5.1 ACCESSIBILITY

The Property can be reached from the City of Williams Lake via Likely Road, which is a paved secondary road that leaves Highway 97 at 150 Mile House, approximately 16 km south of Williams Lake, and continues for 87 km to the community of Likely (Figure 5.1). From Likely, the central and northern part of the Property is accessed from the Spanish Mountain 1300 FSR, which begins east of Likely and continues through the centre of the Property. The southern portion of the claims is accessed from Likely along the Cedar Creek/Winkley Creek Road (FSR 3900), for a distance of about 10 km. Numerous logging roads lie throughout the claim block and offer good access to most areas. A gravel airstrip is located along the 1300 FSR between Kilometres 2 and 3.

## 5.2 CLIMATE

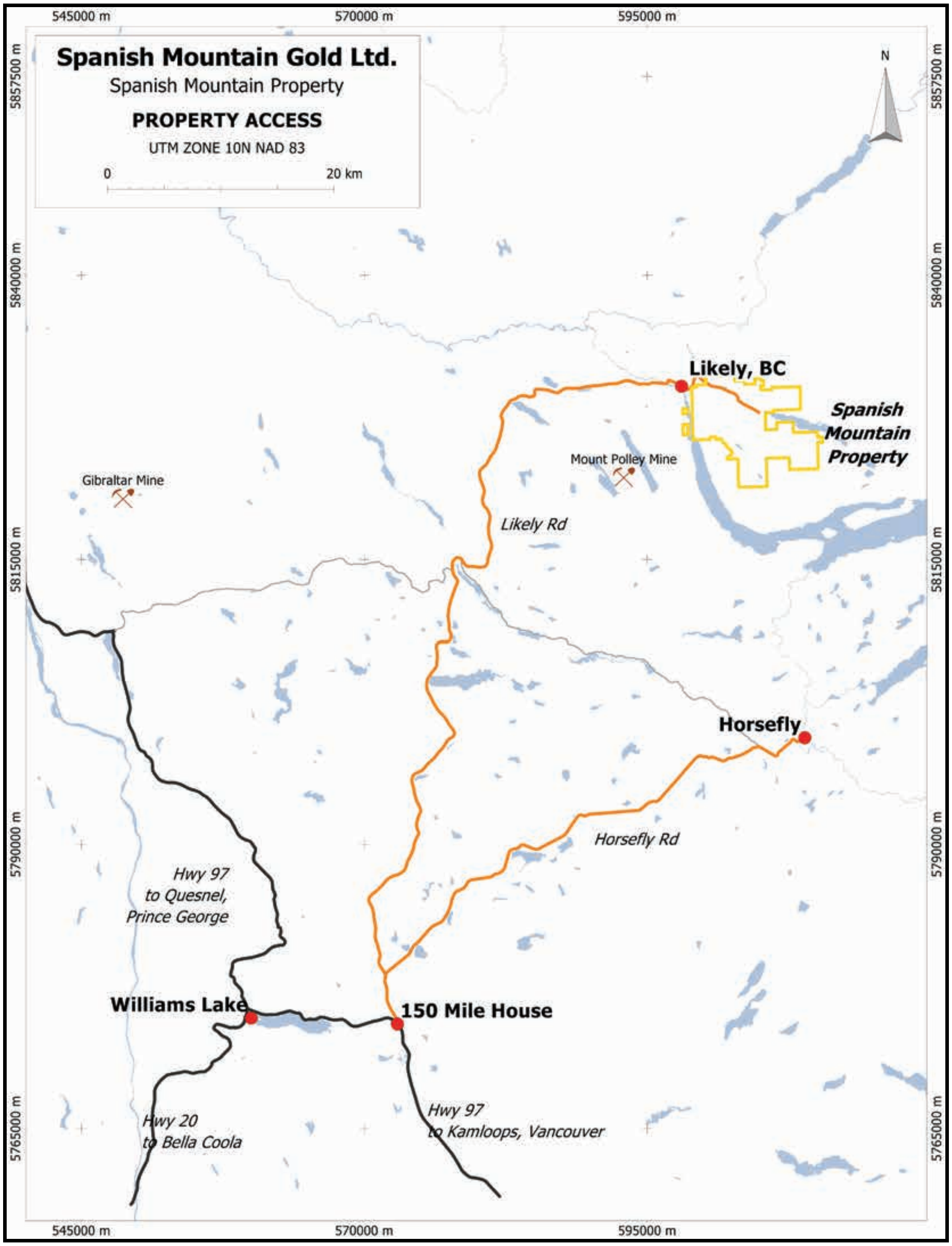
The climate of the Likely area is modified continental with cold snowy winters and warm summers. Likely has an annual average precipitation of approximately 70 cm. Snowfall on the Property is commonly about 200 cm between the months of October and April. Drilling programs can be conducted on a year-round basis, although in the winter months, there are extra costs associated with water availability and road access.

## 5.3 LOCAL RESOURCES

SMG has a modern, full-service facility on purchased land near the Property to provide a base for operations. The community of Likely has basic amenities including a motel, hotel, rental cabins, corner store, gas pumps, and a seasonal restaurant. Some heavy equipment is also available for hire from local contractors. All services and supplies are readily available in Williams Lake, an hour drive from

Likely. The Williams Lake Regional Airport is serviced by three scheduled airlines that provide daily service to Vancouver, BC and points north within BC.

**Figure 5.1 Property Access**



## 5.4 INFRASTRUCTURE

The main access road to the area is the paved Likely Road which links Likely to Highway 97 near Williams Lake. The road is also the primary road used for the mining operation at the Mount Polley open pit copper-gold mine, owned by Imperial Metals Corporation (Imperial Metals). This mine is situated about 15 km in a direct line southwest of the Property. Power is available at Likely, with a major line in place to the Mount Polley mine. Water is abundant in the area.

## 5.5 PHYSIOGRAPHY

The Property covers an area of approximately 9 km north to south by 10 km east to west, situated between Spanish Lake on the east and Quesnel Lake on the west. Physiographically, the area is situated within the Quesnel Highland, which is transitional between the gently undulating topography of the Cariboo Plateau to the west, and the steeper, sub-alpine to alpine terrain of the Cariboo Mountains to the east. The terrain is moderately mountainous with rounded ridge tops and U-shaped valleys. Topography is locally rugged with occasional cliffs and moderately incised creek valleys. Within the Property, elevations range from 910 m at Spanish Lake to 1,582 m at the peak of Spanish Mountain. Drainage is via Spanish Creek, which drains northwest into Cariboo Creek, and Cedar Creek, which drains west into Quesnel Lake. Quesnel Lake flows into Quesnel River, and joined by Cariboo Creek, flows west to eventually join the Fraser River near the town of Quesnel.

Overburden depths are quite variable, ranging from 1 to 10 m in most of the Main Zone, to over 70 m further west in the Cedar Creek area. During the last glacial period, the ice advanced in a northwest direction (Tipper 1971). Rock outcroppings are scarce and are typically found along the crest of ridges, in incised river and creek gullies, and along shore lines.

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

# 6.0 HISTORY

The following section is derived directly from the report "Technical Report on an Updated Mineral Resource Estimate on the Spanish Mountain Gold Deposit", by G. Giroux and A. Koffyberg and dated August 31, 2012. Minor changes have been made for report consistency.

The history of the Property has been summarized by Page (2003), and by Singh (2008). Table 6.1 gives a brief summary of the historical work in tabular form, and has been adapted from Singh (2008) with minor edits. The 2005 to 2009 exploration programs carried out by SMG at that time were done under its former name of Skygold. Work conducted from 2005 to the present is described in more detail in Sections 10.0 and 11.0 of this report.

**Table 6.1 Summary of Historical Exploration**

| Year | Company | Work Completed |
|---|---|---|
| 2009 | SMG | • 13,769 m of diamond drilling in 62 holes. This included drilling in the ROG, Cedar Creek, Placer, North Zone, and Black Bear Mountain areas.<br>• Geological mapping, rock sampling (41 samples).<br>• Soil sampling (121 samples). |
| 2008 | SMG | • 40,449 m of diamond drilling in 161 holes.<br>• Geological mapping, rock sampling, soil sampling. |
| 2007 | SMG | • 26,993 m of diamond drilling in 126 holes.<br>• Metallurgical test work on drill core. |
| 2006 | SMG | • 21,881 m of diamond drilling in 88 holes.<br>• 5,009 m of reverse circulation (RC) drilling in 50 holes.<br>• Geological mapping, rock sampling, soil sampling.<br>• Airborne geophysics and orthophotography on a property-wide scale. |
| 2005 | SMG | • 7,746 m of diamond drilling in 35 holes.<br>• 3,376 m of RC drilling in 30 holes.<br>• Geological mapping, rock sampling, soil sampling. |
| 2004 | Wildrose | • 2,506 m of RC drilling in 34 holes, 2,419 m of trenching, soil sampling.<br>• Discovery of disseminated mineralization in drilling. |
| 2003 | Wildrose | • 30 line km of grid, induced polarization survey (23 line km), soil sampling (1,479 samples), geological mapping. SMG options the Property and begins funding exploration. |
| 2002 | Wildrose | • Small geochemical sampling program. |

*table continues…*

| Year | Company | Work Completed |
|---|---|---|
| 1999-2000 | Imperial Metals | • Imperial Metals options the property and attempts bulk samples from five pits. From one pit, a 1,908 t bulk sample (screened portion of 6,000 t) averages 3.02 g/t gold, based on sampling of 64 truckloads. Blasthole drilling (201 samples from 182 holes) averaged 2.20 g/t gold, based on assays performed at Mount Polley. |
| 1996 | Cyprus Resources Ltd. (Cyprus Resources) | • 2,590 m of trenching signifying the first effort to explore for bulk mineable type disseminated gold mineralization.<br>• 230 m of trench TR96-101 assayed 0.745 g/t gold. |
| 1995 | Eastfield Resources Ltd. | • Optioned the property to Consolidated Logan Mines who then optioned it to Cyprus Resources. |
| 1993-1994 | Cogema Canada Ltd. | • 30 trenches with 900 rock/channel samples. |
| 1992 | Renoble Holdings Inc. | • Stockpiled 635 t from a small open pit in the Madre zone (High Grade zone). The material was processed in two mill runs; 318 t were sent to the Premier Mill (46 troy oz recovered) and 105 t were sent to the Bow Mines Mill (Greenwood, BC) with 105 troy oz recovered. |
| 1992 | Eastfield Resources Ltd. | • Consolidated the property. |
| 1986-1988 | Pundata Gold Corp. | • 37 diamond drillholes (3,273 m), 15 RC holes (1,237 m), 848 m of trenching, geological mapping, sampling (5,350 samples), metallurgical testing of 11 samples, preliminary resource estimate. |
| 1987 | Placer Dome Inc. | • Optioned north and west and south areas of the property. Seven percussion holes (338 m) were drilled; five along the northwest ridge of Spanish Mountain and two near the Cedar Creek drainage. Significant gold values were obtained from overburden section of several holes. |
| 1986 | Mandusa Resources Ltd. | • Optioned the north and southern areas of the property. Conducted geological mapping and IP surveys, and drilled six percussion holes (357 m). |
| 1985 | Mt. Calvery Resources Ltd. (Mt. Calvery Resources) | • Phase 1: 600 m of trenching and sampling, 7 RC holes (655 m).<br>• Phase 2: 820 m of backhoe trenching (550 one-metre channel samples), 29 RC holes (2,521 m). A preliminary resource estimate was made.<br>• Phase 3: 7 diamond drillholes. Teck Resources (Teck) provided funding for Phases 2 and 3. |
| 1984 | Mt. Calvery Resources | • Prospecting, geological mapping, rock and soil sampling including 2,225 m of trenching, 10 diamond drillholes (467 m), 10 RC holes (589 m). |
| 1983 | Whitecap Energy Inc. | • Soil sampling (409 samples) on the CPW claim with values up to 5,100 ppb Au.<br>• 100 m of trenching in 3 trenches. |

*table continues…*

| Year | Company | Work Completed |
|---|---|---|
| 1983 | Lacana Mining Corp. | • Prospecting identified strong gold anomalies coincident with silicified argillite north of Spanish Lake. |
| 1982 | C.P. Wallster | • Staked the CPW claim, as the Mariner II claim had lapsed earlier that year. |
| 1981 | Aquarius Resources Ltd. (Aquarius Resources) | • Soil sampling, airborne geophysical electromagnetic survey. |
| 1979, 1980 and 1982 | E. Schultz, P. Kutney and R.E. Mickle | • Prospecting, sampling, stripping by D-7 and D-8 cats.<br>• 240 m of trenching. Little information is available for this work. |
| 1979 | Aquarius Resources | • Surface exploration and regional assessment of the Likely area. |
| 1977-1978 | LongBar Minerals | • Prospecting (14 rock samples), geological mapping, soil sampling (60 samples) and trenching (14 trenches). |
| 1976 | M.B. Neilson | • Staked the Mariner II claim (High Grade zone). A few samples were collected. |
| 1971 | Spanallan Mining Ltd. | • Magnetometer survey on the Cedar Creek drainage. |
| 1947 | El Toro BC Mines | • 8 drillholes (792 m), 4 tons of handpicked mineralized material shipped to the Tacoma Smelter. |
| 1938 | N.A. Timmins Corp. | • Overburden stripping, drove two small adits on large quartz veins. |
| 1933 | Dickson and Bailey | • Gold discovered in quartz veins on the northwest flank of Spanish Mountain at 1,100 m elevation. |
| 1921 | - | • Placer gold discovered in bench deposits on Cedar Creek. |

# 7.0 GEOLOGICAL SETTING AND MINERALIZATION

The following section is derived directly from the report "Technical Report on an Updated Mineral Resource Estimate on the Spanish Mountain Gold Deposit", by G. Giroux and A. Koffyberg and dated August 31, 2012. Minor changes have been made for report consistency.

## 7.1 REGIONAL GEOLOGY

The Property lies within the Quesnel Terrane of the Intermontane Belt. The rocks of the Quesnel Terrane are predominately sedimentary and volcanic rocks of upper Triassic to early Jurassic in age, representing an island arc and marginal basin assemblage. The eastern boundary of the Quesnel Terrane in the region is marked by the Eureka Thrust, a major southwesterly dipping thrust fault. To the east are the intensely deformed, variably metamorphosed Proterozoic and Paleozoic pericratonic rocks of the Barkerville Subterrane. This includes the Snowshoe Group (unit 7) and the Quesnel Lake Gneiss. Splays of the Eureka Thrust, including the Spanish Thrust, bisect the Spanish Mountain area.

The stratigraphy of the Quesnel Terrane in the Spanish Mountain area has been examined by Rees (1981), Struik (1983), and Bloodgood (1988). Panteleyev et al. (1996) have produced a geological compilation of the Quesnel River – Horsefly area. Nomenclature has varied for the rocks within the central part of the Quesnel Terrane, such as Quesnel River Group, Horsefly Group, Takla Group and Nicola Group; however, Panteleyev et al. assign the term Nicola Group rocks as the most accurate usage. The oldest suite of rocks in the area is the Crooked Amphibolite unit of the Slide Mountain Terrane, of Pennsylvanian to Permian age (unit 6). It consists of talc chlorite schists, amphibolites, serpentinites and ultramafic rocks. This unit is in structural contact with the base of the Quesnel Terrane, and marks the trace of the Eureka Thrust.

The overlying rocks, which belong to the Quesnel Terrane, consist of a sedimentary package of black graphitic argillites, phyllitic siltstones, sandstones, limestones and banded tuffs, (units 5a and 5c), are weakly metamorphosed and belong to the Nicola Group. The age of this unit, based on conodont fossils found south of Quesnel Lake, is Middle to Late Triassic age. A narrow sequence of volcanic and volcaniclastic rocks (unit 5b) occur as a discrete subunit within the sedimentary sequences.

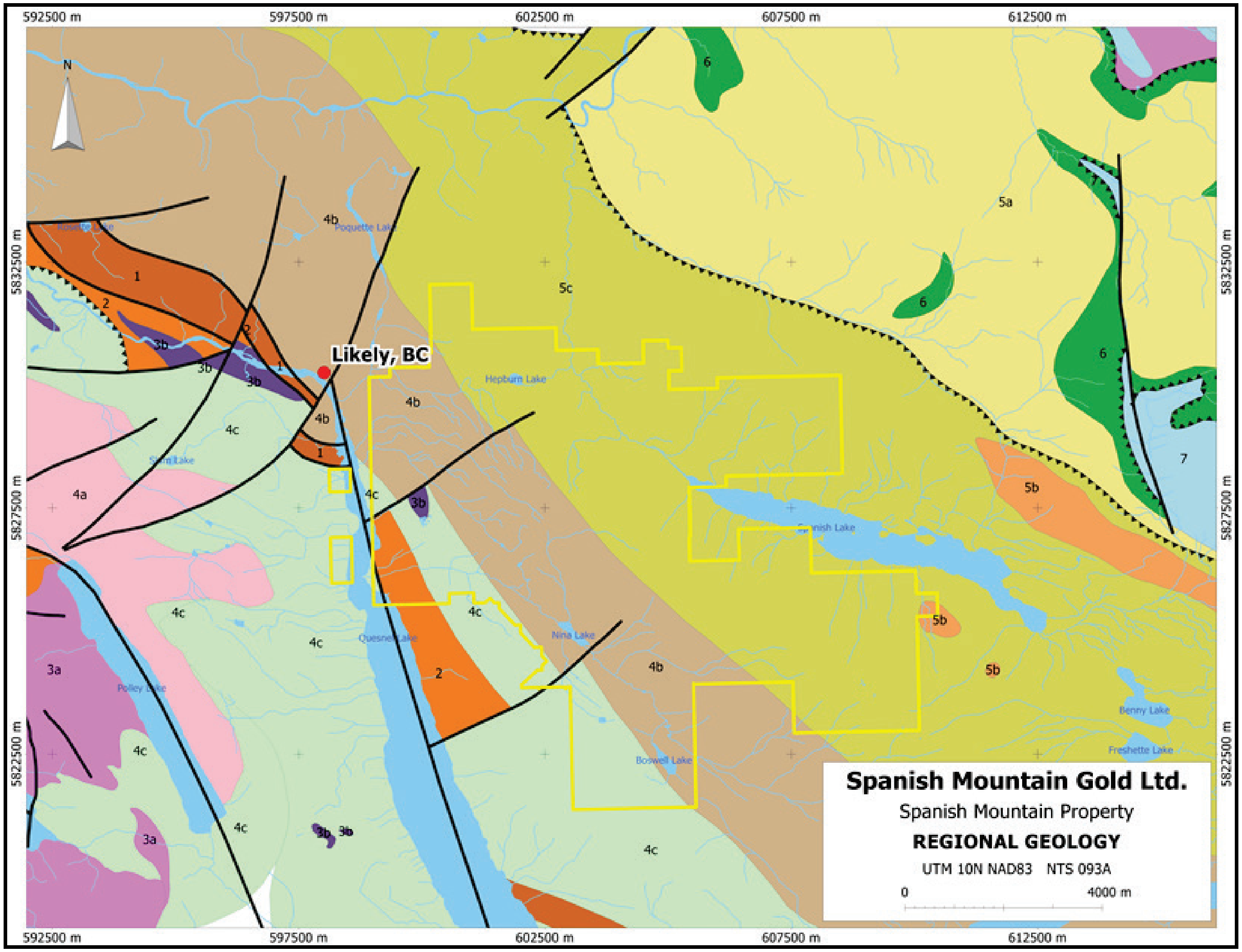
The overlying Nicola Group volcanic rocks (unit 4c) are in depositional contact with the metasediments. The oldest package of volcanic rocks is mainly of alkali composition, and has been divided into an older package of dark grey to green flows, pillow basalts, breccias and tuff, and a younger volcanic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

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

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

Metamorphic mineral assemblages are of sub-greenschist facies. Figure 7.1 shows the regional geology, based on the bedrock geological compilation of the QUEST map area (Logan et al. 2010). The legend is shown in Figure 7.2.

**Figure 7.1 Regional Geology**



### Figure 7.2 Legend of the Regional Geology

**LEGEND**

| SEDIMENTARY, METAMORPHIC & VOLCANIC ROCKS | | INTRUSIVE ROCKS | |
|---|---|---|---|
| **CRETACEOUS** | | | |
| 1 | Undivided sedimentary rocks, conglomerate, sandstone, shale. | | |
| **JURASSIC** | | | |
| LOWER | | | |
| 2 | Argillite, greywacke, conglomerate turbidites; feldspathic sandstone and siltstone; minor limestone and calcareous siltstone. | 3a | Syenite, monzonite, monzodiorite, syenodiorite and diorite; minor nepheline syenite, clinopyroxenite, peridotite and gabbro. |
| **TRIASSIC** | | | |
| UPPER (NICOLA GROUP) | | | |
| 4a | Polymict volcanic breccia containing clasts of latite, trachyte and intrusive equivalents; basalt flows and breccias; some felsic volcanic breccias and flows. | 3b | Syenite, monzonite, monzodiorite, syenodiorite and diorite; minor nepheline syenite, clinopyroxenite, peridotite and gabbro. |
| 4b | Sandstone, siltstone, shale; slate and phyllite; bioclastic limestone; minor felsic tuff, tuffaceous argillite, basalt breccia and agglomerate. | | |
| 4c | Pyroxene and pyroxene-hornblende basalt flows, breccias and tuffs; minor sandstone, siltstone, limestone and limestone breccia. | | |
| MIDDLE (NICOLA GROUP) | | | |
| 5a | Sandstone, siltstone, shale; slate and phyllite; bioclastic limestone; minor felsic tuff, tuffaceous argillite. | | |
| 5b | Pyroxene and pyroxene-hornblende basalt flows, breccias and tuffs. | | |
| 5c | Mixed volcaniclastic rocks, siltstone, sandstone and minor limestone. | | |
| **CARBONIFEROUS-PERMIAN** | | | |
| CROOKED AMPHIBOLITE | | | |
| 6 | Ultramafic rocks- Serpentinite, sheared ultramafic rock, amphibolite, talc schist. | | |
| **UPPER PROTEROZOIC-PALEOZOIC** | | | |
| SNOWSHOE GROUP | | | |
| 7 | Metasediments- quartzite, micaceous quartzite, schist, phyllite, gneiss, marble, amphibolite. | | |

Map is based on the QUEST bedrock map by Logan et al. (2010)

## 7.2 Property Geology

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

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

### 7.2.1 Stratigraphy

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

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

- **Lower Argillite** (with tuffs and siltstone): Black to dark grey, interbedded argillite, tuff and siltstone, with minor felsic dykes. This unit exhibits ankerite and silica alteration and only minor graphite. Pyrite content is generally less than 2%. The unit hosts lesser to minor amounts of gold mineralization.

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

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

The geology of the Property is given on Figure 7.3. The location of the proposed pit was outlined in the 2010 AGP Mining Consultants Inc. (AGP) report. A schematic cross section of the deposit is shown on Figure 7.4, showing the location of the drillholes to 2011.

**Figure 7.3 Property Geology**

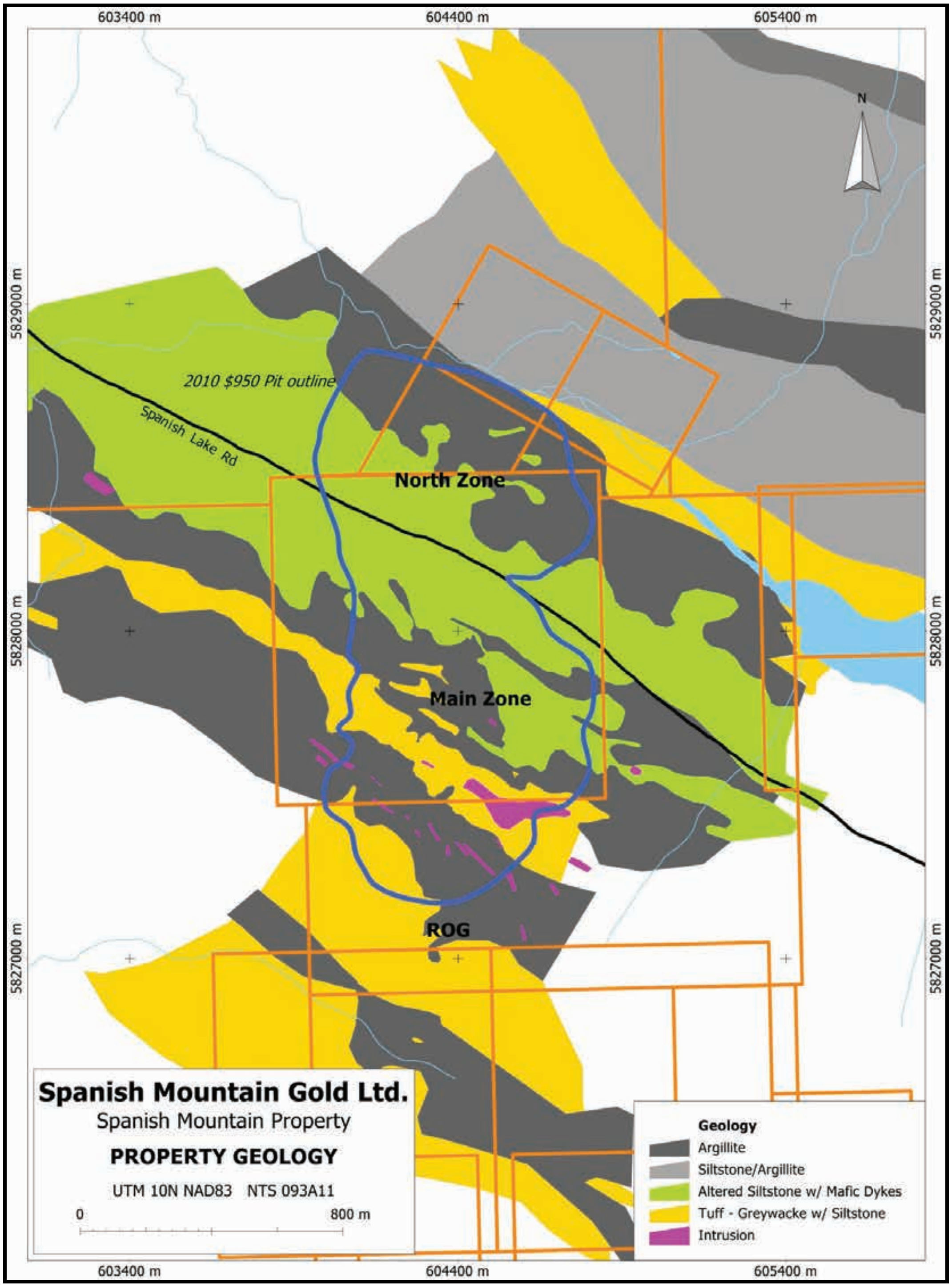
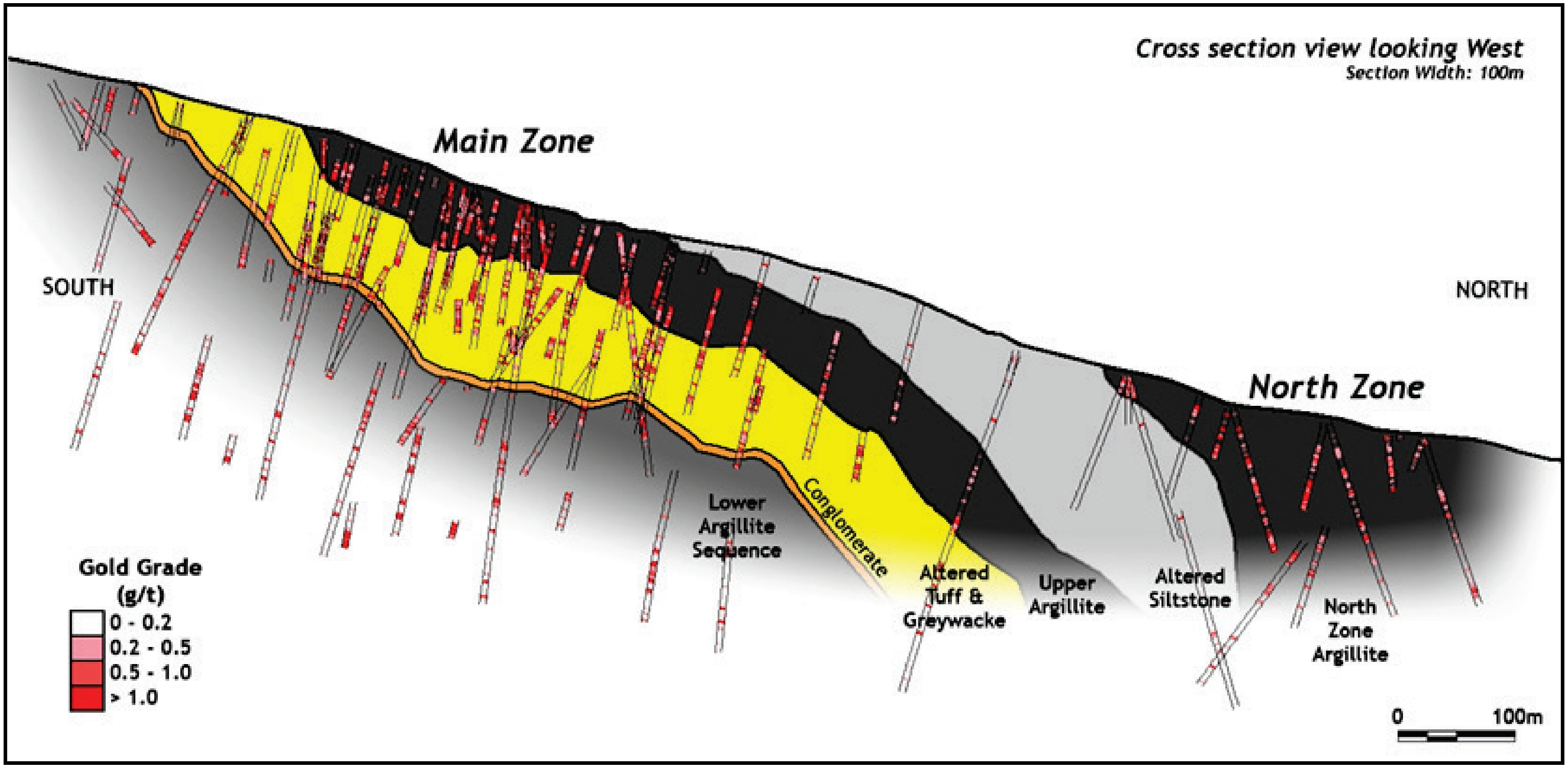


**Figure 7.4 Schematic Cross Section**



### 7.2.2 STRUCTURE

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

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

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

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

### 7.2.3 ALTERATION

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

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

### 7.2.4 MINERALIZATION

Gold mineralization occurs as two main types:

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

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

There is a lack of copper, lead, zinc, arsenic and antimony and other trace metals in the system, and thus the only pathfinder element is gold itself.

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

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

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

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

# 8.0 DEPOSIT TYPES

The following section is derived directly from the report "Technical Report on an Updated Mineral Resource Estimate on the Spanish Mountain Gold Deposit", by G. Giroux and A. Koffyberg and dated August 31, 2012. Minor changes have been made for report consistency.

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

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

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

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

# 9.0 EXPLORATION

The following section is derived directly from the report "Technical Report on an Updated Mineral Resource Estimate on the Spanish Mountain Gold Deposit", by G. Giroux and A. Koffyberg and dated August 31, 2012. Minor changes have been made for report consistency.

This report is concerned primarily with an updated resource estimate for the Main and North Zones and is based on the results of sampling drill core from the programs carried out from 2005 to 2012. Thus a summary is provided of the work done in these programs. Programs carried out before 2005 are summarized in Section 6.0. Note that the 2005 to 2009 exploration programs carried out by SMG at that time were done under its former name of Skygold.

## 9.1 2005 PROGRAM

In 2005, SMG began diamond drilling and continued with RC drilling with joint venture partner Wildrose. A program totalling 7,746 m of diamond core drilling and 3,377 m of RC drilling was carried out, along with geological mapping, rock sampling and soil sampling (Singh 2008).

## 9.2 2006 PROGRAM

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

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

## 9.3 2007 Program

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

## 9.4 2008 Program

A large diamond drilling program consisting of 40,449 m of NQ and NQ2 core in 161 holes was done in 2008 (Peatfield et al. 2009). Drilling focused on the lateral extent of the Main Zone, to the northwest and to the north at depth, and the lateral extent of the North Zone, for a total of 140 holes. Drilling also tested the ROG area where high-grade trench and rock sampling was targeted with 18 drillholes; the Cedar Creek area, where 2 drillholes tested anomalous gold in soils; and the Placer area where 1 drillhole tested an area of an anomalous rock sample.

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

## 9.5 2009 Program

In 2009, definition drilling continued in the Main Zone with a program of 62 core diamond drillholes, totalling 13,769 m (AGP 2010). Of these holes, 33 HQ holes were done on the Main Zone, along with 4 twinned NQ holes, to test whether there was any apparent bias in assay grades in NQ versus HQ size core. The results were inconclusive, since the HQ samples were analysed at a different lab from the NQ samples. In addition, three deep holes were drilled below the Main Zone, ranging in depth from 450 to 650 m, totalling 1,705 m. The holes were collared about 200 m apart along a fence oriented from 119 to 289°. The drillholes intersected thick sequences of sedimentary strata with generally low gold values at depth.

Outside drilling targets were also drilled, including the ROG, Cedar Creek, Placer, North Zone step-out and Black Bear Mountain, for a total of 6,849 m in 21 holes (Montgomery 2009). Other work included reconnaissance geological mapping, rock sampling (41 rock samples) and preliminary re-interpretation of historic data. The Imperial Metals pit and neighbouring trenches on the Main Zone were re-excavated, mapped and chip sampled. A limited soil sampling program was carried out in the ROG area (121 samples) and the Cedar Creek – Mt. Warren area (28 samples).

## 9.6 2010 PROGRAM

The 2010 exploration program consisted of 20 diamond drillholes within and peripheral to the Main and North Zones of the deposit, for a total of 6,834 m (Koffyberg 2011). Seven of the holes were geotechnical holes of HQ3 size within the Main and North Zones. The sites targeted areas of potential waste rock, which will possibly form the pit walls. Four metallurgical (HQ) holes were drilled in the Main and North Zones. These holes were designed to provide information for the on-going metallurgical testing program dealing with gold recoveries. One HQ3 hole, located in the Main Zone, was selected for both geotechnical and metallurgical analysis. The remaining eight NQ holes were exploration holes drilled outside of the boundary of the Main and North Zones, to determine the potential for expansion of the Main/North Zone gold resource.

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

## 9.7 2011 PROGRAM

SMG carried out an infill diamond drilling program on the Main and North Zones, for a total of 82 holes. This work totalled 8,869 m of core diamond drilling from 31 holes in the Main Zone, and 10,568 m of core diamond drilling from 51 holes in the North Zone. The program was designed to provide additional information to enable a re-classification from the Inferred to Measured and Indicated categories. Included in the Main Zone were three deep holes (11-DDH-986,987,988), drilled to test for mineralization at depth. These holes reached depths of 444 m, 566 m and 517 m. One of the holes encountered 23.5 m of 0.58 g/t gold at a depth of 484.5 m; a second hole carried 9.0 m of 1.32 g/t gold at a depth of 489.0 m, indicating that gold mineralization continues with depth. In addition, four of the holes were geotechnical holes, designed to provide information for open pit designs.

A diamond drilling program was undertaken in the North Cedar area where 32 diamond drillholes in a grid-like pattern at intervals of roughly 500 m. Within this area, a new zone of gold mineralization was discovered in late 2011 and termed the Phoenix Zone. This zone is located about 2 km west of the Main Zone. Gold intercepts included 92 m grading 0.58 g/t gold, and 55 m grading 0.82 g/t gold.

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

17 diamond drillholes, resulted in low concentrations of copper over wide intervals, with narrow intervals having higher values over the range of 0.11 to 0.44% copper. Other work included an airborne geophysical survey, which was carried out over the Property in late 2011. This involved a magnetic and DIGHEM V electromagnetic airborne survey, which was carried out by Fugro Airborne Surveys Ltd. Baseline environmental studies continued throughout the year.

## 9.8 2012 PROGRAM

SMG continued definition drilling with an infill diamond drilling program on the Main and North Zones. As of June 18, 2012, the program comprised 131 NQ diamond drillholes, for a total of 24,290 m. This work totalled 19,970 m of diamond drilling from 98 holes in the Main Zone, and 4,320 m of diamond drilling from 33 holes in the North Zone. Subsequent to June 18, 12 holes were drilled, 9 of which were for geotechnical purposes. The results of these holes are not included in the resource estimate in this report and not included in Table 10.1 or on Figure 10.4.

Exploration drilling continued in the North Cedar area to better define the Phoenix Zone, resulting in seven diamond drillholes totalling 2,012 m. Preliminary metallurgical work indicated that the same flowsheet as has been developed on the Main Zone is suitable for the Phoenix Zone.

Baseline environmental studies remain ongoing. The results of the exploration drilling of the Phoenix Zone is outside of the scope of the report and will not be further commented on.

# 10.0 DRILLING

The following section is derived directly from the report "Technical Report on an Updated Mineral Resource Estimate on the Spanish Mountain Gold Deposit", by G. Giroux and A. Koffyberg and dated August 31, 2012. Minor changes have been made for report consistency.

SMG has been drilling on the Property since 2005. Table 10.1 summarizes the drilling activity on the deposit from 2005 onwards by SMG. In total, 670 diamond drillholes (154,368 m) from 2005 to 2012 inclusive have been used in the resource estimate. A complete list of the drillholes is provided in Appendix B.

**Table 10.1 Summary of Drilling Activity by SMG**

| Year | Drill Type | No. of Holes | Metres | Core Size |
|---|---|---|---|---|
| 2012 | Diamond | 131 | 24,290 | NQ |
| 2011 | Diamond | 82 | 19,437 | NQ/HQ3 |
| 2010 | Diamond | 20 | 6,833 | NQ/HQ/HQ3 |
| 2009 | Diamond | 62 | 13,769 | NQ/HQ |
| 2008 | Diamond | 161 | 40,449 | NQ/NQ2 |
| 2007 | Diamond | 126 | 26,993 | NQ |
| 2006 | Diamond | 88 | 21,881 | NQ |
| 2006 | RC | 50 | 5,009 | N/A |
| 2005 | Diamond | 35 | 7,746 | NQ |
| 2005 | RC | 30 | 3,377 | N/A |

For the 2010, 2011 and 2012 programs, diamond drilling was contracted to Atlas Drilling Ltd of Kamloops, BC. Downhole measurements including azimuth and dip were measured using a Reflex EZ-Shot® tool, and were collected every 50 m downhole. Collar locations were initially surveyed using a hand held global positioning system (GPS). The 2010 drill collar locations were later more accurately surveyed by Crowfoot Surveys of Kamloops BC, utilizing standard surveying equipment. Recent 2011 and 2012 surveying was done in-house using Trimble R8R2K Survey GPS equipment supplied by Cansel Survey Equipment Inc. The locations of the 2009, 2010, 2011 and 2012 diamond drillholes are shown in Figure 10.1, Figure 10.2, Figure 10.3 and Figure 10.4, respectively.

Drilling has identified gold mineralization at Spanish Mountain in an area that extends approximately 1,300 m by 800 m. From drillhole data, elevated gold assay results are observed to be laterally continuous along various stratigraphic sequences. The

2011 and 2012 drill programs in particular have expanded the known mineralization in the North Zone.

**Figure 10.1 2009 Drill Locations on the Main and North Zones**

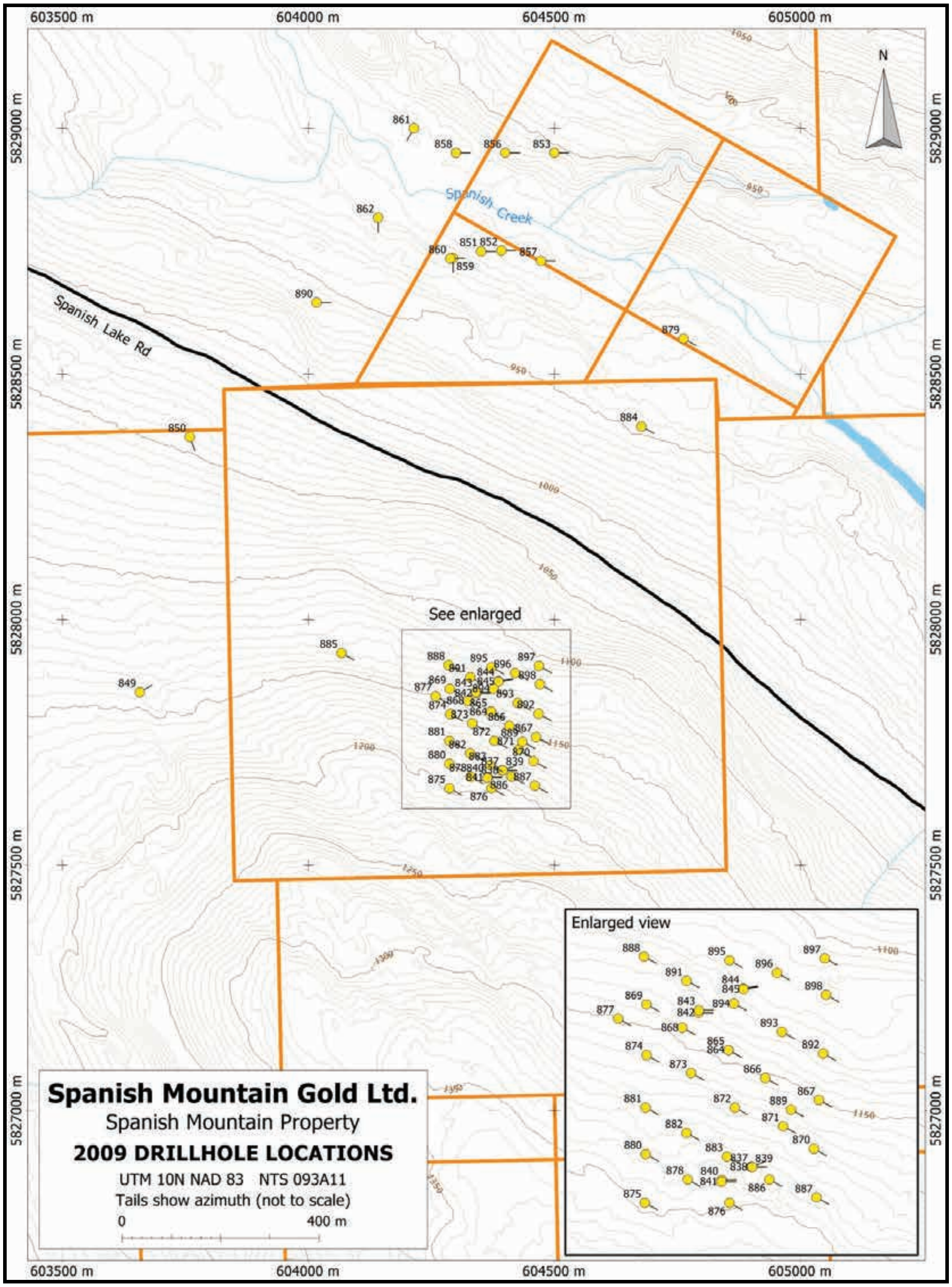


**Figure 10.2 2010 Drill Locations on the Main and North Zones**

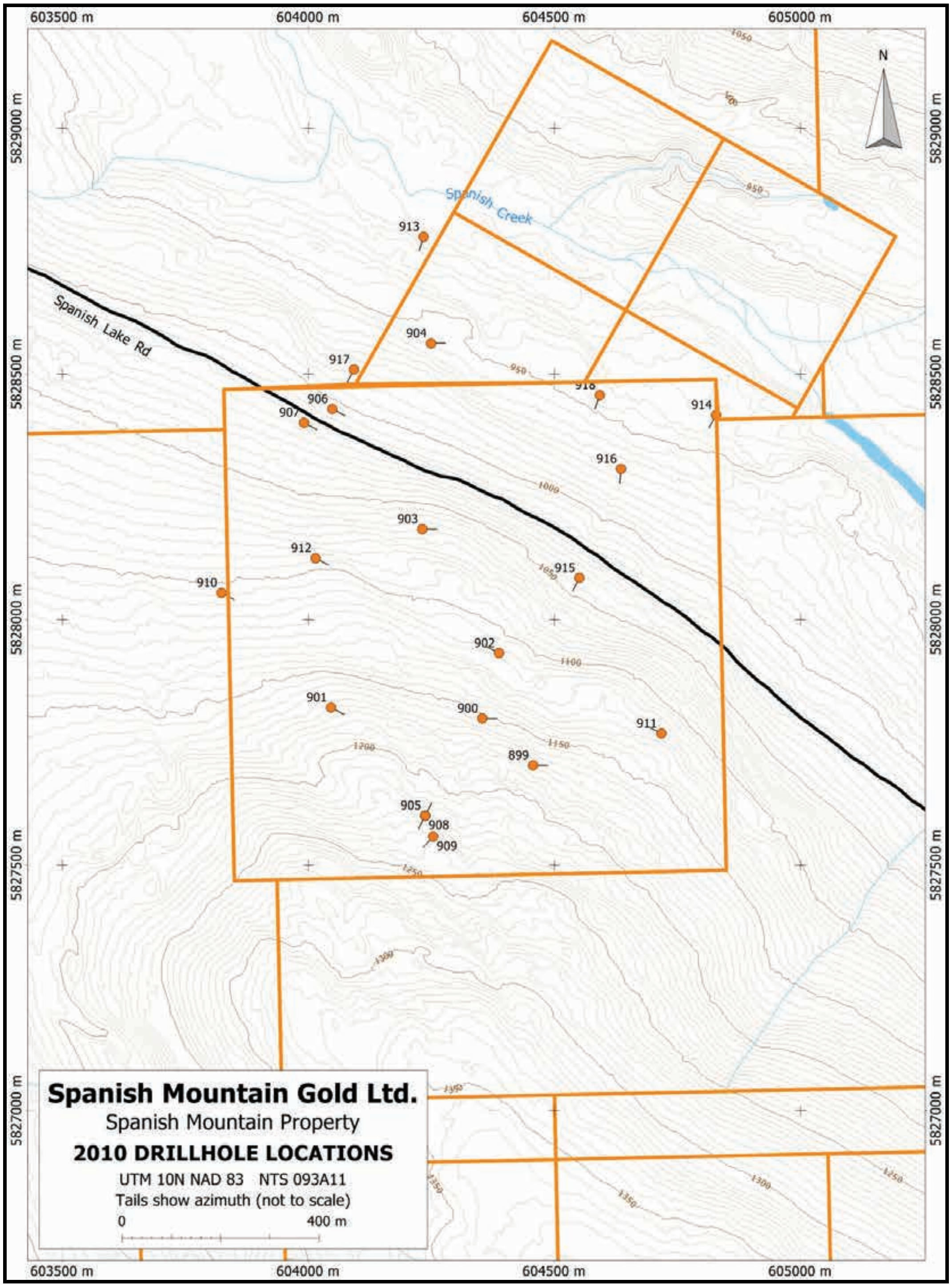


**Figure 10.3 2011 Drill Locations on the Main and North Zones**

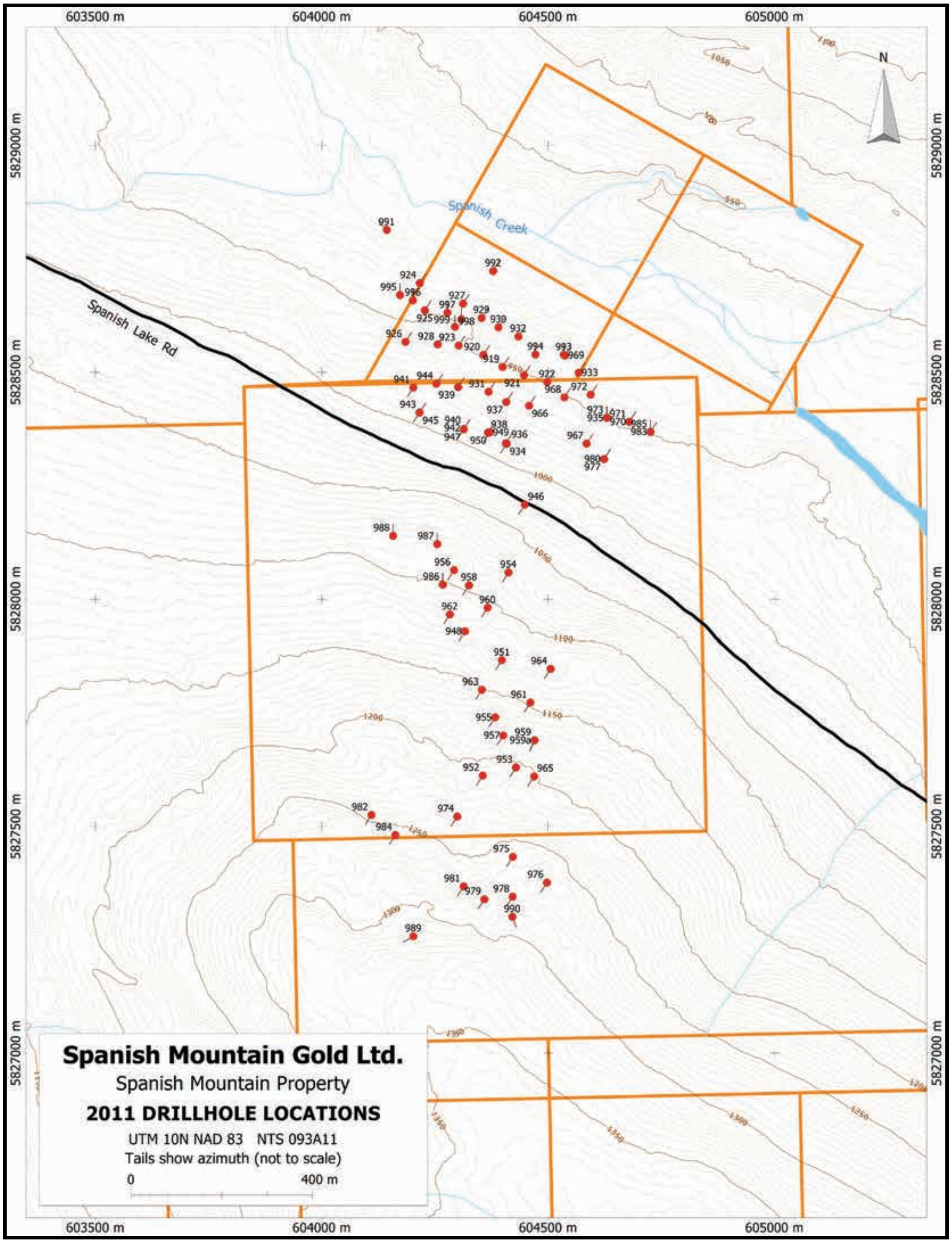
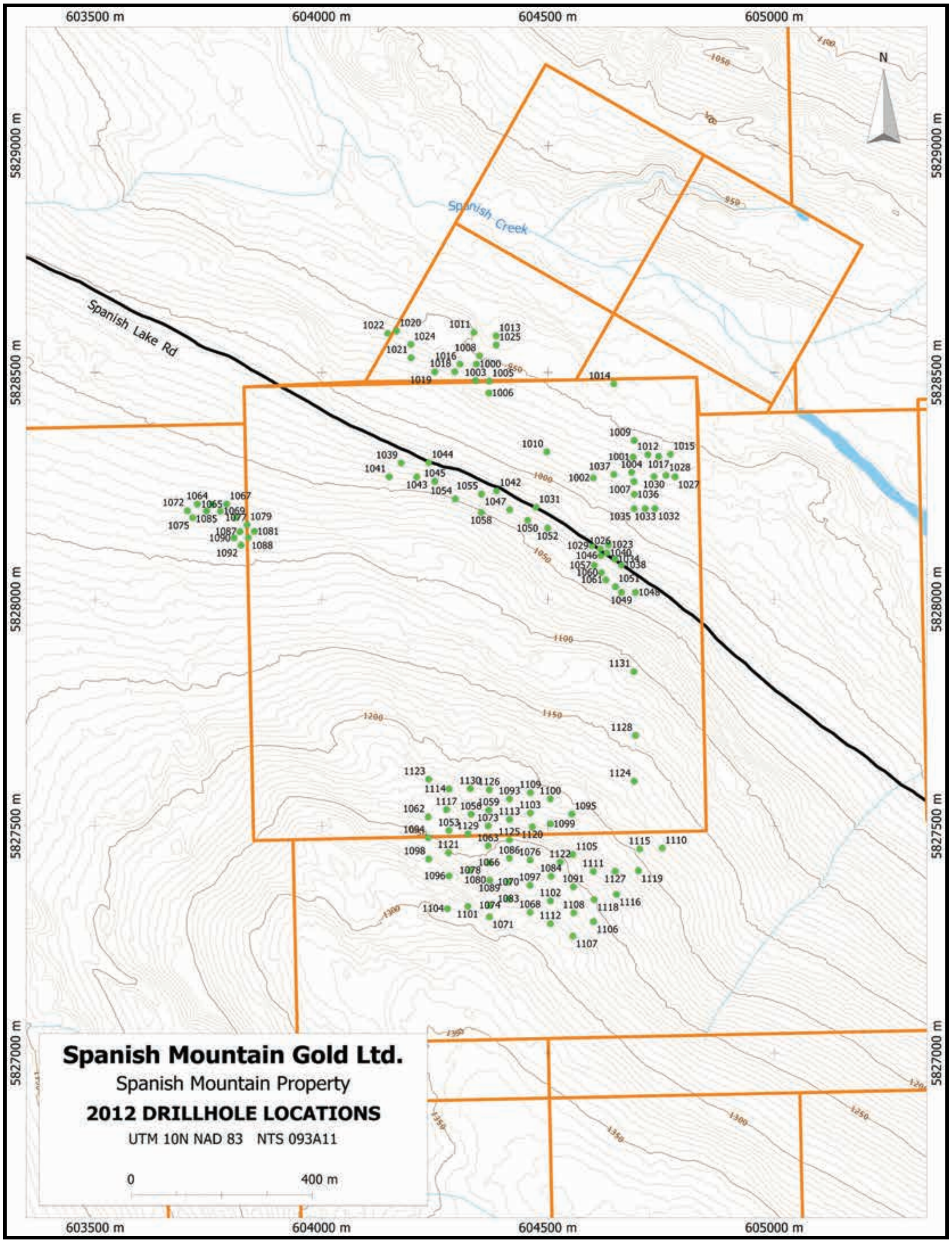


**Figure 10.4 2012 Drill Locations on the Main and North Zones**



# 11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

The following section is derived directly from the report "Technical Report on an Updated Mineral Resource Estimate on the Spanish Mountain Gold Deposit", by G. Giroux and A. Koffyberg and dated August 31, 2012. Minor changes have been made for report consistency.

This section describes the sampling methods used by SMG in 2010, 2011 and 2012 drilling programs. Sampling methods used during the 2005 to 2009 programs are described by Peatfield et al. (2009) and by AGP (2010). The information in this section was obtained from SMG, ALS and a report by W.R. Gilmour (2012), which summarizes a Property visit on April 22, 2012.

Drill core was transported to SMG's core logging facility, where rock quality designation (RQD) procedures, core logging, core splitting and core sampling were done. Also at this facility, blank samples and standards were inserted into the sample stream. This facility is located on SMG's privately-owned property in the community of Likely, located about 7 km from the Main and North Zones. Core storage is also located here.

Core was generally sampled in 1.5 m intervals with shorter lengths given for lithology changes or the presence of visible gold. Core splitting was done using diamond bladed rock saws operated by SMG personnel. Half of the core was sent for analysis; the other half was returned to the core box for a permanent record. Drill core samples were placed in plastic bags and shipped in rice bags through contract personnel (private courier) to ALS in North Vancouver, BC, for sample preparation and analysis. The samples and quality assurance/quality control (QA/QC) samples were tabulated on batch sheets, with every sample batch comprising 80 samples. This ALS facility is certified to standards within ISO 9001:2008 and has received accreditation to ISO/International Electrotechnical Commission (IEC) 17025:2005 from the Standards Council of Canada (SCC).

Analytical procedures used at ALS were:

- gold: fire assay gold, specifically the 1 kg screen metallic method (Au-SCR21), which uses both an atomic absorption finish and a gravimetric finish
- multi-element: four-acid multi-element analysis by inductively coupled plasma-mass spectrometry (ICP-MS) (method ME-ICP61).

The 1 kg screen metallic method involved crushing the entire sample in an oscillating steel jaw crusher for 70% to pass -10 mm. A 1 kg split was pulverized and passed through a 150 mesh (100 µm grain size), producing a plus fraction (i.e. more than 100 µm) and minus fraction (i.e. less than 100 µm). Two 30 g sub-samples of the finer screened material were analyzed by fire assay, with an atomic absorption spectroscopy (AAS) finish. The entire amount of coarser material was also assayed by fire assay, with a gravimetric finish. The gold assays from the two fines were weight averaged, and this assay was then weight averaged with the assay from the coarser fraction, giving an overall assay for the sample.

## 11.1 Quality Assurance/Quality Control Programs

Since December 2011, SMG has retained Discovery of Vernon, BC to independently monitor the QA/QC procedures. The monitoring was done under the supervision of W.R. Gilmour, P.Geo. of Discovery.

QA/QC procedures carried out include the insertion into the sample stream by SMG of:

- field blank samples
- empty bags with sample slips for insertion in ALS's lab of a duplicate reject (prep) samples
- duplicate core samples
- various gold standards (reference material).

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

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

### Collection and Security

The procedures are described in the above section and they are deemed to be satisfactory.

### Contamination

The purpose of field blank sample was to check for contamination within the preparation (crushing, pulverizing) process. Field blanks consisted of sand collected from a gravel pit 30 km west of the Property. These samples, being sand, were not blind to the laboratory. In 2011, each 200 sample batch of blank sand was routinely checked by 15 samples sent for analysis at Eco-tech. This sand was routinely found to be "clean" or devoid of gold mineralization.  The blanks were inserted randomly in the sample stream within every batch of 30 samples.

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

#### PRECISION

Duplicate samples were prepared and analysed to measure precision.  Precision is defined as the percent relative variation at the two standard deviation (95%) confidence level.  In other words, a result should be within two standard deviations of the mean, 19 times out of 20.  The higher the precision number the less precise the results.  Precision varies with concentration – commonly, but not always, the lower the concentration the higher the precision number.  The precision values are determined from Thompson-Howarth plots (Smee 1988).  The duplicate sample results pair the original result with another sub-sample from the core.  Note that the statistical analysis included all 2011 and 2012 data and did not include earlier data.

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

- core sampling
- sample preparation and sub-sampling
- analysis.

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

- core duplicates: the error in the sampling (splitting) of the core, in the sub-sampling of crushed and pulverized core, and in analysis
- reject (prep) duplicates: the error in the sub-sampling of crushed and pulverized core, and in analysis
- pulp duplicates: the error in the sub-sampling of pulverized core, and in analysis.

The duplicates were inserted into the sample stream after the original sample.

##### *Core Duplicates*

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

Sample pairs containing an average grade of at least 0.06 g/t gold (202 pairs) were plotted by the Thompson-Howarth method. These duplicate samples underwent the same metallic gold analysis as did the regular samples. The results are summarized in Table 11.1.

**Table 11.1 2012 Core Duplicates – Precision Values (n = 202)**

| Precision Values (%) | | | | |
|---|---|---|---|---|
| **Au (g/t)** | **0.20** | **0.50** | **0.75** | **1.00** |
| Au | 42.2 | 83.6 | 92.8 | 97.4 |

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

*Reject (or Prep) Duplicates*

For the 2011 drilling used in the 2011 resource estimate, the laboratory systematically produced, every 30 samples (901 pairs), another sample from the saved reject (crushed) core. Sample pairs containing an average grade of at least 0.04 g/t gold (418 pairs) were plotted by the Thompson-Howarth method. These duplicate samples underwent the standard fire assay gold analysis on only the -150 mesh pulp. The results are summarized in Table 11.2.

**Table 11.2 2011 Reject Duplicates – Precision Values (n = 418)**

| Precision Values (%) | | | | |
|---|---|---|---|---|
| **Au (g/t)** | **0.20** | **0.35** | **0.50** | **0.75** |
| Au | 41.6 | 36.3 | 34.3 | 32.6 |

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

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

Sample pairs containing an average grade of at least 0.06 g/t gold (209 pairs) were plotted by the Thompson-Howarth method. The results are summarized in Table 11.3.

**Table 11.3 2012 Reject Duplicates – Precision Values (n = 418)**

| Precision Values (%) | | | | |
|---|---|---|---|---|
| **Au (g/t)** | **0.20** | **0.50** | **0.75** | **1.00** |
| Au | 31.6 | 27.0 | 26.0 | 25.4 |

At the 95% confidence level the precision values indicate about a ±16% error for 0.20 g/t gold values, about a 14% error for 0.50 g/t gold, and about a ±13% error for 1.00 g/t gold values. This is the total error for sub-sampling of crushed core (reject or prep) and pulverized core, and analysis.

### *Pulp Duplicates*

For the 2010, 2011 and 2012 drilling, ALS prepared two 30 g sub-samples per sample for every sample of core, producing 15,317 pairs. Sample pairs containing an average grade of at least 0.04 g/t gold (7,278 pairs) were plotted by the Thompson-Howarth method. The results are summarized in Table 11.4.

**Table 11.4 Pulp Duplicates – Precision Values (n = 7278)**

| Precision Values (%) | | | | |
|---|---|---|---|---|
| **Au (g/t)** | **0.20** | **0.50** | **0.75** | **1.00** |
| Au | 48.6 | 23.4 | 18.3 | 15.6 |

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

## ACCURACY

All but one of the SMG inserted gold standards were produced by CDN Resource Laboratories Ltd. (CDN) of Langley, BC, and were certified to two standard deviations by a certified assayer and a professional geochemist. One standard was produced by Ore Research & Exploration of Australia.

Standards have been analysed throughout the drill programs from 2005 to 2012. In the 2010 and 2011 drill programs, one of three standards was added randomly to a batch of 30 samples. For the 2010 drilling, standards were submitted with expected grades of 0.39, 0.78, 1.16 and 4.83 g/t gold; and for the 2011 drilling, standards had expected grades of 0.21, 0.39, 0.78, 1.14, 1.16 and 3.77 g/t gold.

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

The quality assurance monitoring of the results included plotting the results for each SMG and ALS standard in order of report completion. The charts were regularity reviewed for results outside of the expected values ranges. Occasionally re-analysis of a group of samples was done. However, for the 2012 drill program, no changes in the results were warranted.

It is Discovery's opinion that the sample security, sample preparation and analytical procedures during the exploration programs by SMG followed accepted industry practice appropriate for the stage of mineral exploration undertaken, and are NI 43-101 compliant.

# 12.0 DATA VERIFICATION

The following section is derived directly from the report "Technical Report on an Updated Mineral Resource Estimate on the Spanish Mountain Gold Deposit", by G. Giroux and A. Koffyberg and dated August 31, 2012. Minor changes have been made for report consistency.

The 2005 drilling program by SMG was under the supervision of Robert Darney, P.Geo. of Pamicon Developments Ltd (Pamicon).

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

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

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

# 13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

## 13.1 INTRODUCTION

In 2010, AGP Mining Consultants Inc. (AGP) completed a PEA-level study, which included a historical review of test work conducted from 2007, and included the results of the more recent test work completed in 2010. Between 2010 and July 2012, additional test work on a series of composite and individual samples was conducted at two independent laboratories: G&T Metallurgical Services (G&T), based in Kamloops, BC and SGS Minerals Services (SGS), based at their facility in Lakefield, ON. Results from a gravity concentration test work program, and a subsequent modelling of the results obtained, were reported by Knelson Research and Technology Centre (Knelson Research), based in Langley, BC.

Based on the metallurgical test work results, Tetra Tech adopted a conventional grinding, gravity concentration and flotation circuit design. The gravity concentrate will be processed by intensive cyanidation leaching, while the flotation concentrate will be processed by conventional CIL technology. The final product will be doré bars containing gold and silver. The results from all the test work have provided the basis for the process and flowsheet design and development.

The gold concentrator for the Project has been designed to process a nominal 14,600,000 t/a, or 40,000 t/d, of gold and silver bearing material from an open pit operation, and will produce gold-silver doré as a final product. The process description is provided in Section 17.0.

## 13.2 METALLURGICAL TEST WORK REVIEW

Sample material was collected from drillhole material and prepared for metallurgical testing. These samples were subject to a series of test work programs, including:

- comminution test work
- whole-ore cyanidation leach tests
- gravity concentration tests
- flotation tests
- gravity concentration/flotation of gravity tailings test work
- cyanidation of gravity middlings and recombined middlings/tailings test work.

The results obtained are described in the following sections.

### 13.2.1 CHRONOLOGY OF TEST PROGRAMS AND REPORTS

The test work programs and reports listed in Table 13.1 are a chronology of all the test programs conducted on the SMG resource to date. Further testing to define the pit zone and for variability testing is currently underway in order to generate sample material.

**Table 13.1 Test Work Programs and Reports**

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

*table continues…*

| Document or Test Program | Author or Laboratory | Date |
|---|---|---|
| Gravity Modelling Report – Spanish Mountain Gold, KRTC 20559-1 | Knelson Research | October 18, 2011 |
| A Variability Test Program on Samples from the Spanish Mountain Deposit, Project 12488-002-Report #2 | SGS | March 19, 2012 |
| Metallurgical Testing on Variability Samples from the Spanish Mountain Gold Project, Report No. KM3185 | G&T | June 21, 2012 |

### 13.2.2 MINERALOGY

The SMG deposit is described as a gold-based sediment-hosted vein deposit. The gold present in the SMG deposit occurs as free gold associated with quartz veins and as attachments to and occlusions in pyrite. The deposit contains carbonaceous material, graphite, which has been shown to be "preg-robbing". The nature of the association of gold with the carbonaceous material has not been reported.

A petrographic study was performed in 2006 to determine the characteristics of the gold mineralization found in diamond drill core intervals. The gold grade of the samples ranged from 0.2 to 3.46 g/t gold. The samples were variably carbonaceous siltstone/mudstones and included fine grained greywackes. In some instances, up to 30% of the mineralization was carbonaceous material. Native gold was identified in four samples as inclusions and fracture-fill in pyrite on crystal boundaries between pyrite crystals and in the gangue adjacent to pyrite. The particles were very fine-grained—less than 20 µm and generally less than 5 µm—and were described as occurring in 15 of the 21 samples studied. There was no clear indication from the study whether the gold was preferentially associated with any particular habit of pyrite, or other mineral type.

Little information regarding the nature the mineralogical evaluations from the deposit has been documented. The G&T report KM1921 conducted mineralogical studies on three master composite samples. The chemical composition of the samples showed a scarcity of minerals containing copper, lead, zinc, arsenic, antimony and other trace elements. Carbon was found to be present; an average of 22% of the carbon present occurring was in organic form.

Gold mineralization was noted and described in the AGP report as follows:

- free gold in quartz veins
- free gold in fractures in pyrite
- free gold in base metal sulphides (sphalerite, chalcopyrite, galena)
- gold associated with arsenopyrite

- disseminated gold in black argillite material
- disseminated gold in fault structures often associated with quartz veins
- nuggets measuring up to 100 mm across which have been collected by placer miners over the years. This is likely an example of a style 1 occurrence, i.e. gold in quartz veins. Several pieces of gold of up to approximately 3 mm in size have been observed in drill cores.

### 13.2.3 HEAD GRADE

The head grades of various test samples have been characterized by assay and the results have been detailed in a number of reports. The variation in the sulphur content and total organic carbon (TOC) content, as well as the gold grade, are factors which give indications of the mineral composition of the sample.

The G&T report KM1921 presented the results of 13 composite samples compiled from various drill cores and tested. These samples were then combined to create three master composite samples and one master composite blend sample. The head assays of these samples ranged from 0.82 g/t gold to 7.48 g/t gold. The complete data can be located in the test report. The feed compositions of the four master composite samples, based on assay, are shown in Table 13.2.

**Table 13.2 Head Composition – G&T**

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

Carbon and TOC assays were carried out on the three composite samples. In some instances, a significant portion of the carbon occurred in the organic form. In particular, the composite sample SM12 contained 2.7% TOC which comprised 87% of the total carbon in the sample. Only two of the original 13 samples which were used to make up the composites had less than 20% of the total carbon contained in organic form.

The G&T report KM2538 was a test program designed to determine the gold content of 148 core intervals using mineral processing to minimize the effect of nugget-bearing gold using gravity concentration. G&T planned to include a comparison of these results, with results obtained by conventional analysis in the report, but this aspect of the test program was not completed because conventional assay results were not received by the laboratory. The main emphasis was placed on the gravity recovery of the gold in the samples, which is discussed in Section 13.2.6. The results of the metallurgical assays gave head assays which ranged from 0.02 to 6.20 g/t gold.

The G&T progress report KM2637 also had three master composite samples created from one drillhole and this was used for a defined metallurgical test program. This drillhole was located in the starter pit area of the deposit. The gold and TOC grades varied in the samples which allowed for variation in the samples for testing purposes. The gold grades were lower than for the previous master composites, although the iron and sulphur values were reasonably similar. Table 13.3 gives the assay values obtained for the sample material tested.

**Table 13.3 Feed Composition – G&T**

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

Variability testing was conducted at two laboratories: G&T and SGS. The variability composite sample assays for gold, TOC and sulphur are reported in Table 13.4 and Table 13.5 respectively.

**Table 13.4 Variability Composite Assays – G&T**

| Composite ID | Assays | | |
|---|---|---|---|
| | Au (g/t) | TOC (%) | S (%) |
| 871 | 1.05 | 1.09 | 3.30 |
| 3 | 0.30 | 0.14 | 0.86 |
| 4 | 0.10 | 0.03 | 0.24 |
| 5 | 0.19 | 0.49 | 0.72 |
| 6 | 1.17 | 1.12 | 2.58 |
| 7 | 0.27 | 0.12 | 0.70 |
| 8 | 0.57 | 0.07 | 0.52 |
| 9 | 0.27 | 0.98 | 1.07 |
| 10 | 0.54 | 1.40 | 2.09 |
| 12 | 0.43 | 0.66 | 2.21 |
| 13 | 0.17 | 1.61 | 3.67 |
| 15 | 0.03 | 0.06 | 0.28 |
| 16 | 0.10 | 0.14 | 0.51 |
| 17 | 0.06 | 0.04 | 0.22 |

*table continues…*

| Composite ID | Assays | | |
|---|---|---|---|
| | Au (g/t) | TOC (%) | S (%) |
| 18 | 0.48 | 1.55 | 4.14 |
| 19 | 1.68 | 1.53 | 3.70 |
| 20 | 1.27 | 1.71 | 3.55 |
| 21 | 0.20 | 0.79 | 0.59 |
| 22 | 0.03 | 0.04 | 0.20 |
| 23 | 0.15 | 1.59 | 2.04 |
| 928-A | 0.38 | 1.22 | 3.18 |
| 928-B | 0.29 | 1.79 | 3.14 |
| 928-C | 0.71 | 0.70 | 2.43 |
| 928-D | 0.44 | 1.66 | 2.15 |
| 981 | 0.48 | 2.03 | 3.90 |
| Mixed Composite 25 | 0.25 | 1.81 | 4.55 |
| **Average** | **0.45** | **0.89** | **1.95** |

**Table 13.5 Variability Composite Assays - SGS**

| Composite ID | Assays | | |
|---|---|---|---|
| | Au (g/t) | TOC (%) | S (%) |
| 1 | 0.40 | 0.74 | 0.81 |
| 2 | 0.91 | 1.39 | 2.56 |
| 3 | 0.29 | 0.84 | 0.84 |
| 6 | 1.38 | 1.08 | 2.16 |
| 7 | 0.31 | 0.48 | 0.96 |
| 8 | 0.24 | 0.71 | 0.39 |
| 9 | 0.38 | 1.09 | 0.81 |
| 10 | 0.31 | 1.53 | 1.95 |
| 11 | 0.10 | 0.59 | 0.18 |
| 12 | 0.33 | 0.83 | - |
| 14 | 0.94 | 0.95 | 2.37 |
| 18 | 0.43 | 1.57 | 3.68 |
| 19 | 1.88 | 1.70 | 3.10 |
| 20 | 0.81 | 1.52 | 2.03 |
| 21 | 0.29 | 1.40 | - |
| 24 | 0.42 | 1.86 | 1.86 |
| **Average** | **0.60** | **1.10** | **1.69** |

Samples 3, 7 and 8 are characterized as predominantly tuff, the remaining samples being either predominantly argillite or a mixture of argillite and siltstone, with the exception of Sample 11 which is a waste sample of crystal tuff. Disregarding the waste sample, the SGS gold values ranged between 0.24 and 1.88 g/t gold, and averaged 0.60 g/t gold; TOC values ranged between 0.48 and 1.57%, and averaged 1.69% TOC. These values are generally within the range of the master composite samples used for the test work programs.

The G&T equivalent values varied more widely; values ranged between 0.03 and 1.68 g/t gold and averaged 0.45 g/t gold value and TOC values ranged between 0.03 and 2.03% TOC and averaged 0.89% TOC. The range of sulphur values between 0.20 and 4.55% is typical for these variability samples. However, the G&T samples show that there is a greater proportion of low gold grade material present in these samples.

### 13.2.4 Physical Characteristics

There are two main mineral-bearing rock types identified through testing: argillite and a general non-argillite classification. The design was based on a 50/50 split but the two rock types do not occur in equal in abundance with 156 Mt argillite and 49 Mt non-argillite material. The non-argillite material includes the rock types siltstone and tuff/greywacke. The two rock types are differentiated by the concentration of organic carbon present in the rock with the former generally containing more than 0.5% TOC, and the latter less than 0.5% TOC. Typical density values for the different rock types are shown in Table 13.6.

**Table 13.6 Rock Type Density Values – SGS**

| Argillite | | Tuff | | Siltstone | | Crystal Tuff | |
|---|---|---|---|---|---|---|---|
| **Sample ID** | **Relative Density** | **Sample ID** | **Relative Density** | **Sample ID** | **Relative Density** | **Sample ID** | **Relative Density** |
| SM-1 | 2.72 | SM-3 | 2.72 | SM-5 | 2.72 | SM-11 | 2.75 |
| SM-2 | 2.77 | SM-4 | 2.79 | SM-12 | 2.76 | SM-15 | 2.80 |
| SM-6 | 2.76 | SM-7 | 2.77 | SM-23 | 2.71 | SM-16 | 2.80 |
| SM-9 | 2.72 | SM-8 | 2.77 | SM-24 | 2.72 | SM-17 | 2.77 |
| SM-10 | 2.71 | SM-22 | 2.78 | - | - | - | - |
| SM-13 | 2.76 | - | - | - | - | - | - |
| SM-14 | 2.74 | - | - | - | - | - | - |
| SM-18 | 2.75 | - | - | - | - | - | - |
| SM-19 | 2.74 | - | - | - | - | - | - |
| SM-20 | 2.73 | - | - | - | - | - | - |
| SM-21 | 2.67 | - | - | - | - | - | - |
| **Average** | 2.73 | - | 2.77 | - | 2.73 | - | 2.77 |
| **Standard Deviation** | 0.03 | - | 0.03 | - | 0.02 | - | 0.02 |
| **Minimum** | 2.67 | - | 2.72 | - | 2.71 | - | 2.75 |
| **Maximum** | 2.77 | - | 2.79 | - | 2.76 | - | 2.80 |
| **75$^{th}$ Percentile** | 2.76 | - | 2.78 | - | 2.73 | - | 2.78 |
| **No. of Samples Tested** | 11 | - | 5 | - | 4 | - | 4 |

The density value of the two predominant rock types, using a 50/50 split, averaged 2.75. A value of 2.76 was used in the design criteria of the process design which is the average of the 75$^{th}$ percentile values.

### 13.2.5 GRINDABILITY

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

Subsequently, during August 2010, 24 composited variability samples taken from drillholes across the deposit, were processed at SGS. The variability samples were classified as the various rock domains (i.e. argillite, tuff, siltstone, and crystal tuff) that would be processed and collectively termed non-argillite.

Twenty-four samples were used in the following grindability tests:

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

The SGS test results, including the results for the G&T samples, have been compiled into Table 13.7, along with the data for the different rock type domains.

## Table 13.7 Summary of Grindability Results by Rock Type – SGS and G&T

| Argillite | | | | | Tuff | | | | | Siltstone | | | | | Crystal Tuff | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sample** | **RWi** | **BWi** | **Ai** | **CWi** | **Sample** | **RWi** | **BWi** | **Ai** | **CWi** | **Sample** | **RWi** | **BWi** | **Ai** | **CWi** | **Sample** | **RWi** | **BWi** | **Ai** | **CWi** |
| SM-1 | 12.4 | 10.9 | 0.224 | 9.1 | SM-3 | 14.1 | 12.9 | 0.213 | 14.4 | SM-5 | 16.1 | 14.7 | 0.264 | 11.5 | SM-11 | 17.5 | 16.7 | 0.275 | 18.0 |
| SM-2 | 13.2 | 11.9 | 0.111 | 11.4 | SM-4 | 15.1 | 13.3 | 0.251 | 14.7 | SM-12 | 16.7 | 15.7 | 0.271 | 14.6 | SM-15 | 17.0 | 16.0 | 0.188 | 11.6 |
| SM-6 | 13.8 | 12.5 | 0.226 | 12.0 | SM-7 | 15.3 | 13.5 | 0.197 | 14.7 | SM-23 | 14.1 | 15.9 | 0.281 | 12.8 | SM-16 | 16.3 | 15.1 | 0.214 | 17.0 |
| SM-9 | 14.4 | 13.3 | 0.215 | 10.5 | SM-8 | 15.1 | 13.3 | 0.215 | 13.4 | SM-24 | 14.2 | 15.3 | 0.259 | 11.6 | SM-17 | 16.0 | 14.5 | 0.298 | 15.1 |
| SM-10 | 14.8 | 13.8 | 0.298 | 11.7 | SM-22 | 13.9 | 12.1 | 0.120 | 12.4 | - | - | - | - | - | - | - | - | - | - |
| SM-13 | 13.3 | 13.0 | 0.200 | 8.9 | 865-1 | - | 11.4 | - | - | - | - | - | - | - | - | - | - | - | - |
| SM-14 | 12.9 | 13.3 | 0.173 | 11.1 | 865-3 | - | 12.3 | - | - | - | - | - | - | - | - | - | - | - | - |
| SM-18 | 12.7 | 12.6 | 0.281 | 9.9 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| SM-19 | 12.4 | 12.7 | 0.282 | 10.6 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| SM-20 | 13.8 | 13.7 | 0.299 | 12.9 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| SM-21 | 13.8 | 12.9 | 0.207 | 11.4 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| 865-2 | - | 12.6 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| **Average** | 13.4 | 12.8 | 0.229 | 10.9 | - | 14.7 | 12.7 | 0.199 | 13.9 | - | 15.3 | 15.4 | 0.269 | 12.6 | - | 16.7 | 15.6 | 0.244 | 15.4 |
| **Standard Deviation** | 0.79 | 0.79 | 0.06 | 1.22 | - | 0.65 | 0.78 | 0.05 | 1.00 | - | 1.32 | 0.53 | 0.01 | 1.44 | - | 0.68 | 0.97 | 0.05 | 2.82 |
| **Minimum** | 12.4 | 10.9 | 0.111 | 8.9 | - | 13.9 | 11.4 | 0.120 | 12.4 | - | 14.1 | 14.7 | 0.259 | 11.5 | - | 16.0 | 14.5 | 0.188 | 11.6 |
| **Maximum** | 14.8 | 13.8 | 0.299 | 12.9 | - | 15.3 | 13.5 | 0.251 | 14.7 | - | 16.7 | 15.9 | 0.281 | 14.6 | - | 17.5 | 16.7 | 0.298 | 18.0 |
| **75th Percentile** | 13.8 | 13.3 | 0.282 | 11.5 | - | 15.1 | 13.3 | 0.215 | 14.7 | - | 16.3 | 15.8 | 0.274 | 13.2 | - | 17.1 | 16.2 | 0.281 | 17.2 |
| **No. of Samples Tested** | 11 | 12 | 11 | 11 | - | 5 | 7 | 5 | 5 | - | 4 | 4 | 4 | 4 | - | 4 | 4 | 4 | 4 |

The following is a discussion relative to the data presented in Table 13.7. The deposit consists approximately of 50% argillite and 50% non-argillite rock types which consist mainly of the rock type tuff. The siltstone component will not exceed 5% based on the current mine plan. Although the crystal tuff sample is the hardest and most abrasive of the samples, the mill design does not include the crystal tuff material, since those samples were all below the expected gold cut-off grade as stipulated by SMG.

Overall, the CWi values ranged from 8.9 to 18.0 kWh/t. The argillite samples are generally a soft material with an average CWi value of 10.9 kWh/t, while the Tuff had an average CWi value of 13.9 kWh/t. The average of the 75$^{th}$ percentile results for the two samples described above is 13.1 kWh/t. This CWi value has been used in the design of the process.

The RWi values ranged from 12.4 to 17.5 kWh/t, while the BWi values ranged from 10.9 to 16.7 kWh/t. The average values obtained based on rock type showed that the argillite consumes the least power for grinding; the crystal tuff consumes the most. The BWi value for Tuff was similar to that of the argillite, while the value for siltstone was similar to that of crystal tuff. The average value of the 75$^{th}$ percentile results for the RWi for the two domain samples is 14.5 kWh/t; for BWi, it is the corresponding 75$^{th}$ percentile 13.3 kWh/t. These values were used as the basis for the design of the grinding circuit.

**Table 13.8 Bond Work Index Results – G&T**

| Sample | BWi (kWh/t) |
|---|---|
| Composite 928-A | 13.1 |
| Composite 928-B | 13.2 |
| Composite 928-C | 13.5 |
| Composite 928-D | 13.7 |
| Composite 939-A | 12.6 |
| Composite 939-B | 13.7 |
| Composite 981 | 13.0 |
| **Average** | 13.3 |
| **75$^{th}$ Percentile** | 13.6 |

The results obtained for more recent testing for grindability in the KM3185 test program indicates that the BWi values fall within design range chosen as shown in Table 13.8. The 75$^{th}$ percentile average value of 13.6 kWh/t obtained is slightly higher in this case when compared with the value of 13.3 kWh/t which was used in the design.

The Ai values obtained range from 0.111 to 0.299 g, which classifies the abrasiveness of the samples as mild to medium. The 75$^{th}$ percentile average value is 0.269 g; this value was used in the design of the comminution circuit.

In addition to the work index determinations, SGS carried out JKTech drop weight tests, or SMC tests, on each composite. These tests provide information about the resistance to breakage of the sample material by impact and abrasion. This series of tests also indicated that the softest of the rock types was the argillite, based on the obtained Axb. The results also indicated that when a semi-autogenous grinding (SAG) mill is used for grinding, a pebble crusher will be required in closed circuit with the SAG mill. In its 2010 report, SGS recommended blending the material because of the variance in the material hardness of the various rock types. The complete test results are located in SGS's 2010 report.

### 13.2.6 GRAVITY CONCENTRATION

Various gravity concentration test work programs were conducted during the various metallurgical test programs.

The recovery of gold by gravity methods from the SMG deposit has been evaluated, and so-called gravity gold recovery percentages of up to the lower 50s were reported. Gravity recovered concentrate generally was found to have a lower amount of carbon associated with it, and as such the gold recovery via leaching has been relatively high. This has been demonstrated in the studies conducted to date, with up to 98.6% gold recovery realized from gravity concentrates after regrinding.

Gravity concentration results from the G&T Reports KM2538 and KM2637 were extensively analyzed by SMG and presented in an internal report "Spanish Mountain Gold – Process Development Summary". In summary, the gravity gold was not easily characterized and the results could not be used as a basis for determining the actual amount of gold recoverable by gravity concentration in a commercial plant operation. The average recovery of gold to the gravity concentrate in this test work was 42% for the non-argillite samples, and 26.3% for the argillites, or 34.1% as an overall average. However, a more detailed review by Tetra Tech indicates that the following notes should be applied to the methodology leading to these recovery numbers.

- The collected gravity concentrate is actually a set-specific volume. Therefore, if the feed amount varies, the recovery to the concentrate will also vary regardless of feed grade.
- Gravity recovery is dependent on background gangue and feed size, as well as the specific gravity of the background minerals.
- Production concentrate grade values from commercial operations are not achievable in a laboratory setting, and cannot be related (as mentioned above).
- Every sample has a unique trade-off between the fineness of grind and the recovery of gold to the gravity product, and the optimal gravity recoverable gold size value may be different from the overall primary grind size.

However, any unrecovered gravity gold will still have an opportunity for recovery in the flotation circuit.

- With a batch gravity operation, the sulphide mineral recovery will be very limited since the mass recovered is typically less than 0.05% of the feed. Therefore, not all the gold associated with the sulphide minerals will be recovered but rather more of the free or liberated gold together with some associated gold with pyrite particles.

In 2010, Knelson Research was provided with two composite samples for Extended Gravity Recoverable Gold (EGRG) testing. The EGRG test procedure consists of sequential grinding and recovery stages in order to establish the amenability of the material to gravity concentration. Two different samples from the centre of the Main Zone were provided for this program and the head sample results are given in Table 13.9.

**Table 13.9 Feed Grade of Sample for EGRG Test – Knelson Research**

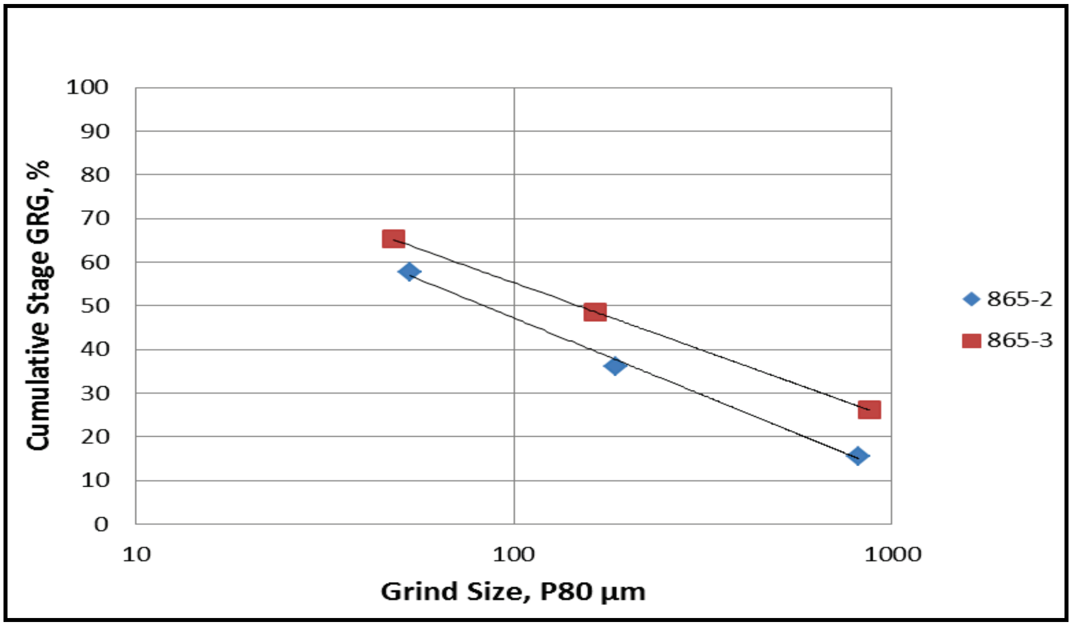
| Composite ID | Rock Type | Assays | | |
|---|---|---|---|---|
| | | Au (g/t) | TOC (%) | S (%) |
| 865-2 | Argillite | 0.94 | 1.18 | 2.96 |
| 865-3 | Rhyolite Tuff/Greywacke | 0.82 | 0.26 | 1.40 |

A summary of the recovery results obtained for different particle sizes is given in Table 13.10, and the cumulative results have been are provided in Figure 13.1.

**Table 13.10 Overall Gravity Recovery Results – Knelson Research**

| Product | Sample Composite 865-2 | | | | Sample Composite 865-3 | | | |
|---|---|---|---|---|---|---|---|---|
| | Grind Size (µm) | Weight (%) | Au (g/t) | Distribution (%) | Grind Size (µm) | Weight (%) | Au (g/t) | Distribution (%) |
| Concentrate Stage 1 | 819 | 0.3 | 54.6 | 15.6 | 873 | 0.3 | 76.9 | 26.1 |
| Concentrate Stage 2 | 186 | 0.3 | 74.4 | 20.6 | 164 | 0.3 | 65.2 | 22.5 |
| Concentrate Stage 3 | 53 | 0.3 | 80.5 | 21.6 | 48 | 0.3 | 56.1 | 16.6 |
| **Total** | **-** | **0.9** | **69.6** | **57.8** | **-** | **0.9** | **66.5** | **65.2** |

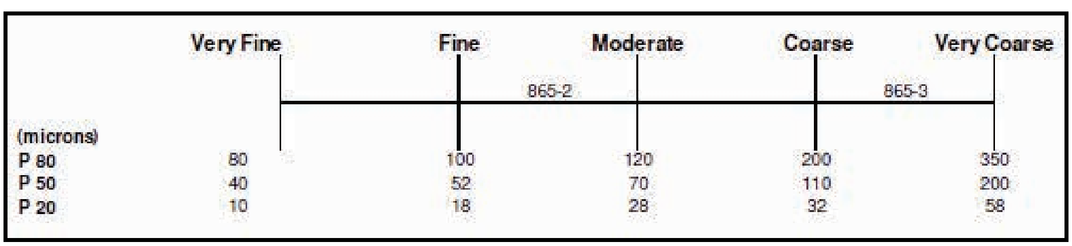
**Figure 13.1 Cumulative Three-stage EGRG Results – Knelson Research**




Analysis of the results obtained by Knelson Research indicates that a nugget of gold was recovered in the 425 to 600 µm size fraction of the 865-3 non-argillite sample, and the results were subsequently revised to account for this event (as shown in Figure 13.3)

A semi-quantitative analysis indicated that the gold in the argillite sample would be classified as fine to moderate on the Amira P420B scale, and the gold in the sample of non-argillite classified as coarse to very coarse, as shown in Figure 13.2. This classification only refers to the sample tested and only to the gold recovered, and did not use the revised results given in Figure 13.3.

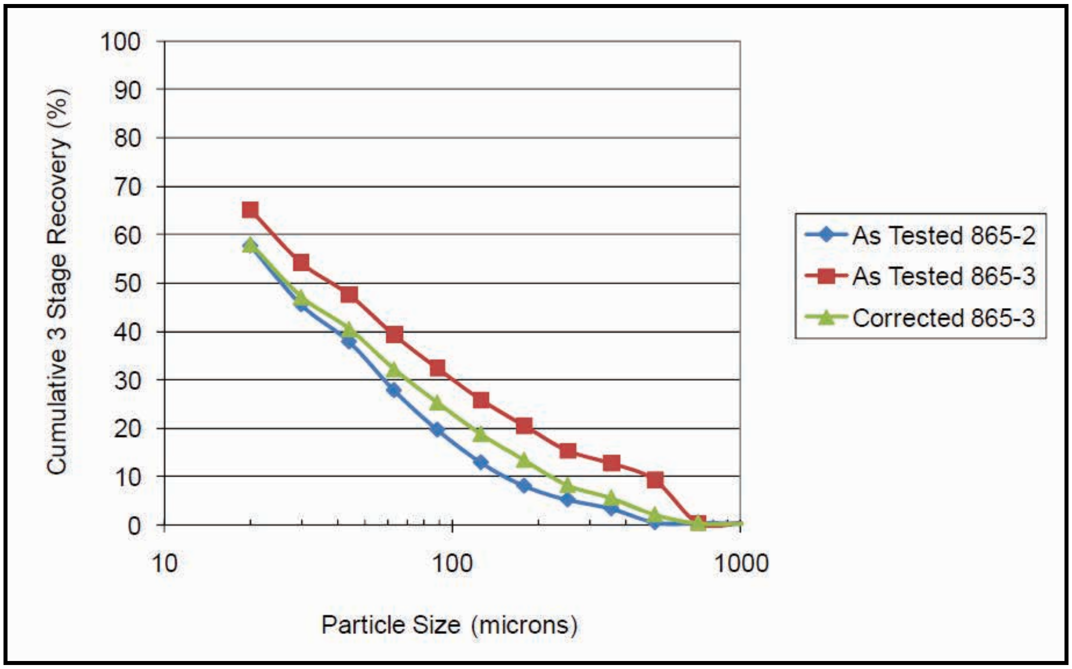
**Figure 13.2 Concentrate Gold Grain Size Classification Scale – Amira P420B**




A modelling report based on the results shown in Table 13.10 was requested from Knelson Research and was provided as report KRTC 20559-1. A mathematical model was used to predict the actual gold recoverable from the grinding circuit.

As mentioned, initially the results in Table 13.10 were normalized to account for the gold nugget in sample 865-3, and the “corrected” classification curve was generated and is shown in Figure 13.3 and Table 13.11.

**Figure 13.3 Cumulative Three-stage EGRG Results (Revised) – Knelson Research**




From the revised curves, the anticipated EGRG values at various grind sizes are presented and shown in Table 13.11. The results indicate that there will be no significant variation in gravity recoverable gold for the two rock types tested.

**Table 13.11 EGRG Adjusted for Plant Grind Size – Knelson Research**

| Grind ($P_{80}$ µm) | Sample No. EGRG (%) | |
|---|---|---|
| | 865-2 | 865-3 |
| 175 | 38.4 | 40.2 |
| 125 | 43.6 | 44.8 |
| 100 | 47.0 | 47.8 |

The data from Table 13.11 was used in the modelling program together with anticipated treatment rates in the plant. At the time that this evaluation was conducted, a mill feed rate of 22,500 t/d was assumed as this was the anticipated throughput rate as provided by SMG. The results for the anticipated plant gravity circuit gold recoveries are shown in Table 13.12.

**Table 13.12 EGRG Values at Various Treatment Rates and Grind Sizes – Knelson Research**

| Gravity Feed Rate (t/h) | Grind $P_{80}$ 175 µm | | Grind $P_{80}$ 125 µm | | Grind $P_{80}$ 100 µm | |
|---|---|---|---|---|---|---|
| | 865-2, Au Recovery (%) | 865-3, Au Recovery (%) | 865-2, Au Recovery (%) | 865-3, Au Recovery (%) | 865-2, Au Recovery (%) | 865-3, Au Recovery (%) |
| 300 | 7.9 | 10.4 | 10.3 | 13.0 | 10.8 | 13.3 |
| 600 | 11.8 | 14.9 | 14.9 | 18.4 | 15.6 | 19.0 |
| 900 | 14.4 | 17.8 | 18.0 | 21.6 | 18.8 | 22.3 |
| 1200 | 16.4 | 20.0 | 20.2 | 23.6 | 21.3 | 24.8 |
| 1500 | 18.1 | 21.7 | 22.1 | 25.7 | 23.3 | 26.7 |
| 1800 | 19.5 | 23.1 | 23.6 | 27.1 | 24.9 | 28.3 |
| 2100 | 20.7 | 24.2 | 24.9 | 28.4 | 26.4 | 29.6 |
| 2400 | 21.8 | 25.3 | 26.1 | 29.4 | 27.6 | 30.8 |
| 2700 | 22.8 | 26.2 | 27.1 | 30.4 | 28.7 | 31.8 |
| 3000 | 23.8 | 27.0 | 28.0 | 31.2 | 29.8 | 32.7 |

Note: Based on 22,500 t/d throughput

The primary design grind size $P_{80}$ is 184 µm, and therefore the 175 µm modelling results were used in the design for the gravity circuit equipment selection which gives an average recovery of 21.3% for a 1,800 t/h treatment rate using 6 gravity concentrator machines. This recovery value has been used in the design of the process plant although the gravity circuit has subsequently been modified to incorporate eight gravity concentrator machines at a combined feed rate of 2,400 t/h.

### Impact of Gravity Concentration on Overall Gold Recovery

There are several impacts regarding the inclusion of gravity concentration to the overall recovery of gold. First, it is anticipated that the recovery to a combined gravity concentrate plus bulk flotation concentrate will be greater than the recovery to a bulk flotation concentrate without gravity concentration. Second, due to the very low carbon content of the gravity product, the separate cyanidation of the gravity and flotation concentrates could result in a greater gold extraction than cyanidation of the combined products, although this is deemed to be a minor advantage. Third, the gravity circuit will contribute towards the “smoothing out” of flotation feed and CIL operations. Fourth, there may be a slight reduction in the usage in reagents.

An additional major impact is that gold particles are unfloatable because they are too coarse or coated with iron oxide alteration products, will be recovered by gravity concentration. Since placer mining has been practiced on a major scale for many years in this area, there is a high probability that coarse gold particles do exist in the deposit.

While there may not be a consistent or dramatic increase in gold recovery to the combined gravity and flotation concentrate compared to flotation without gravity, on balance the inclusion of gravity is justified even though the benefits may not be directly measurable in the residue assays because of the low assay values which will be attained during normal plant operations.

Although gravity concentration has been included in the design, as mentioned previously, an analysis of relevant test work results have not confirmed that there will be a benefit in the overall recovery of gold.

Typically test work was conducted with flotation subsequent to the gravity concentration stage. There were only a limited number of tests conducted without the incorporation of a gravity step. The most relevant results are shown in Table 13.13 for comparison purposes.

Other than the gravity stage, there were variations in the test procedure which impacted the final outcome and recovery. An example is the case of Composite 2. Test 7 had insufficient reagent addition which contributed to the poor flotation recovery. Once the reagents were adjusted in Test 8, results of flotation only as compared to gravity plus flotation of Test 9 show no significant advantage attributable to having the gravity concentration step included.

Based on the limited test data available, gravity concentration shows no clear benefit on the basis of the test work conducted. As indicated, mass recovery and gravity recovery values have the most significant impacts on gold recovery. However, as previously stated, gravity concentration was included in the design of the process plant.

**Table 13.13 Gold Recovery Including and Excluding Gravity Concentration – G&T**

| Composite | Test No. | Comments Test Type | Head Grade Au (g/t) | Flotation Feed Grade Au (g/t) | Rougher Concentrate Mass Recovery (%) | Rougher Concentrate Grade Au (g/t) | Rougher Recovery + Gravity Recovery (%) | Rougher Flotation % Recovery (Flotation only) |
|---|---|---|---|---|---|---|---|---|
| 1 | 6 | No gravity Rougher Flotation | 0.47 | 0.47 | 7.4 | 6.06 | 96.0 | 96.0 |
| | 11 | Gravity Rougher Flotation | 0.44 | 0.28 | 8.8 | 3.04 | 97.9 | 96.8 |
| 2 | 7 | No gravity Rougher Flotation | 0.92 | 0.92 | 8.8 | 9.03 | 86.2 | 86.2 |
| | 8 | No gravity Rougher Flotation | 1.01 | 1.01 | 16.1 | 6.02 | 96.6 | 96.6 |
| | 9 | Gravity Rougher Flotation | 0.98 | 0.67 | 12.1 | 4.91 | 92.3 | 88.4 |
| 3 | 5 | No gravity Rougher Flotation | 0.89 | 0.89 | 8.9 | 9.57 | 96.2 | 96.2 |
| | 3 | Gravity Rougher Flotation | 0.88 | 0.47 | 8.9 | 8.85 | 96.9 | 94.3 |
| | 49 | Gravity – Pre-flotation + Rougher Flotation | 1.21 | 0.59 | 9.4 | 5.91 | 96.3 | 92.9 |
| 891 | 52 | Gravity – Pre-flotation Rougher Flotation | 1.03 | 0.76 | 10.2 | 7.01 | 96.0 | 94.6 |
| | 55 | Gravity – Pre-flotation + Rougher Flotation | 1.22 | 1.10 | 9.0 | 11.38 | 93.9 | 93.2 |
| | 56 | No gravity – Pre-flotation + Rougher Flotation | 1.10 | 1.10 | 11.6 | 8.88 | 93.7 | 93.7 |
| 4 | 58 | Gravity – Pre-flotation + Ro Flotation | 0.96 | 0.74 | 6.8 | 10.3 | 95.4 | 94.1 |
| | 64 | No Gravity – Pre-flotation + Rougher Flotation | 0.87 | 0.87 | 9.7 | 9.9 | 92.5 | 92.5 |

### GRAVITY SCAVENGING OF PRE-FLOTATION CONCENTRATE AND CLEANER TAILINGS

A unique portion of the gold recovery design is the removal of the recycle streams within the flotation circuit. This concept has been developed in order to minimize the effect of typical recycle streams in the process particularly in reducing the recirculation of deleterious carbonaceous material. Tetra Tech recognized that an amount of gold was being lost in the pre-flotation concentrate and the cleaner flotation tailings streams. One method of reducing the losses while minimizing the mass to be recovered, while rejecting carbonaceous material, is through the use of a centrifugal gravity concentration device. It must be noted that this test program was completed prior to the inclusion of the reagent carboxymethyl cellulose (CMC) in all stages of the test program, and the possible effect of CMC is therefore not known.

A composite sample of pre-flotation concentrate and another sample of combined cleaner tailings were prepared from the available products of a series of metallurgical tests. A Knelson Research gravity concentrator was used for the test work. Due to the relatively small amount of sample available for testing, the results of these tests are indicative only and are not intended to provide definitive design data. The results of these tests are summarized in Table 13.14.

**Table 13.14 Results of Gravity Testing on Flotation Reject Streams – G&T**

| Composite Identification | Product | Weight % Distribution | Assays | | | Distribution | | |
|---|---|---|---|---|---|---|---|---|
| | | | Au (g/t) | S (%) | TOC (%) | Au (g/t) | TOC (%) | S (%) |
| Pre-flotation Concentrate | Feed | 100.0 | 0.93 | 0.65 | 2.71 | - | - | - |
| | Gravity Concentrate | 1.6 | 26.60 | 4.81 | 1.25 | 45.2 | 11.7 | 0.7 |
| | Gravity Tailings | 98.4 | 0.52 | 0.59 | 2.73 | 54.8 | 88.3 | 99.3 |
| Cleaner Flotation Tails | Feed | 100.0 | 1.56 | 4.44 | 0.82 | - | - | - |
| | Gravity Concentrate | 5.2 | 13.20 | 23.80 | 0.66 | 44.6 | 28.1 | 4.2 |
| | Gravity Tailings | 94.8 | 0.91 | 3.37 | 0.83 | 55.4 | 71.9 | 95.8 |

The results indicate that gravity concentration is capable of scavenging about half of the contained gold into an enriched stream while rejecting a significant portion of the TOC. The mass of concentrate recovered in the test work is significantly higher than what would actually be recovered in an operating plant. However, this does not mean that, in practice, there would be proportionally less gold recovered. With the additional use of CMC in the various flotation circuits, the TOC content of the feed to the gravity scavenger circuit is expected to increase. The test work indicates that some gold can be recovered by this method, approximately 45% according to the test work results, although the extraction of gold from this product has not been quantified. In the design of the process plant, the secondary gravity concentrate recovered from the combined pre-flotation concentrate and the tailings from the cleaner flotation stages, will also be delivered to the CIL feed thickener and added to the flotation concentrate and the primary gravity concentration tailings for subsequent gold extraction.

### 13.2.7 FLOTATION

Since gravity has been incorporated into the design and into the test program, the analysis of the flotation test work will disregard the effect of gravity on the overall recovery and instead look at the parameters which enhance the flotation portion of the program independently. The parameters that will be analyzed include the incorporation of grind size, pre-flotation, reagent type and addition, flotation time, and cleaning of the rougher concentrate.

The main objective of the flotation circuit is to achieve maximum gold recovery and rejection of TOC prior to the cyanidation circuit. In order to achieve this objective, stage-wise testing was conducted to reduce the amount of TOC.

From the time that is was recognized that SMG material was refractory and "pre-robbing", flotation has been employed to upgrade the gold contained within the sulphide minerals, since direct cyanidation could not be employed. In 1997, G&T investigated the cyanidation of both the whole sample material, commonly referred to as “whole-ore” cyanidation tests, and of specific flotation products. Various conclusions were drawn from these studies, which are discussed in the cyanidation section.

The basic flowsheet employed in the 2010 PEA utilized rougher flotation but incorporated two stages of cleaning to produce a product for the leaching of the gold in a CIL circuit. The second cleaning stage was used to reduce the TOC, which was required to be below 1.0% and preferably below 0.5% in order for efficient leaching to occur. The flowsheet also did not incorporate the recirculation of cleaner tailings to avoid a build-up of carbonaceous material.

The mass recovered to the rougher flotation, the incorporation of pre-flotation and the final inclusion of CMC to each stage of the design all have interacting implications in the overall recovery realized and additional work will be required for quantification.

#### GRIND

The original test work in the test program KM1921 had indicated that a grind finer than $P_{80}$ of 100 µm would be required to produce a recovery of greater than 90%. A detailed review of a subsequent test program using only flotation recoveries has indicated that the gold recovery to a rougher concentrate did not appear to be grind-dependent in the range tested (i.e. a $P_{80}$ value between 97 and 272 µm), and that a grind size of $P_{80}$ of 184 µm is optimal as indicated in Figure 13.4 and Figure 13.5. This grind size $P_{80}$ of 184 µm was used in the subsequent test work and has been incorporated into the overall design of the process plant. A gold flotation recovery of close to 95% was achieved in all cases for 8 minutes of laboratory flotation time, except where the grind was extremely coarse (i.e. $P_{80}$ value of 272 µm).

**Figure 13.4 Primary Grind Size versus Flotation Kinetics – G&T**

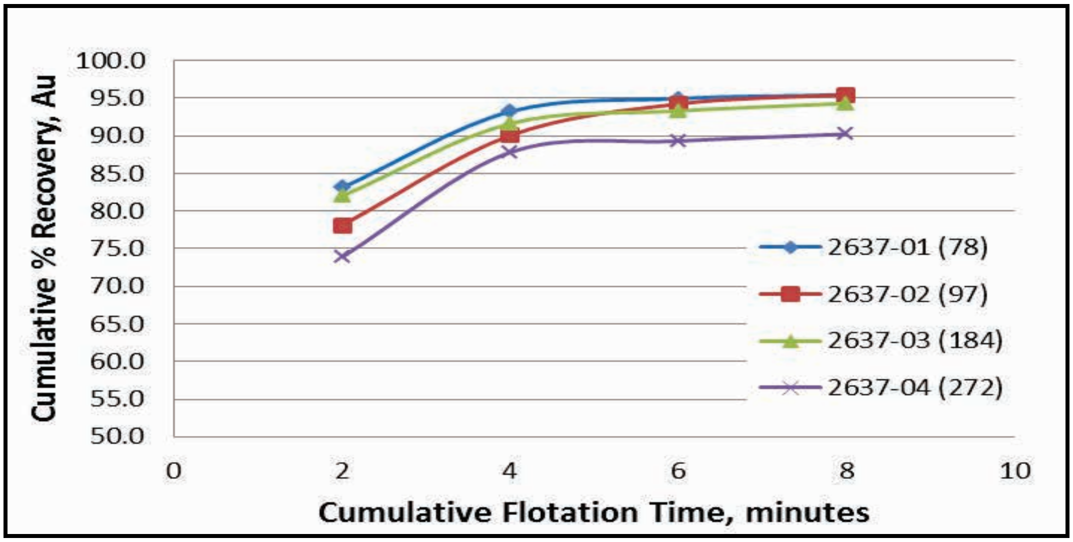
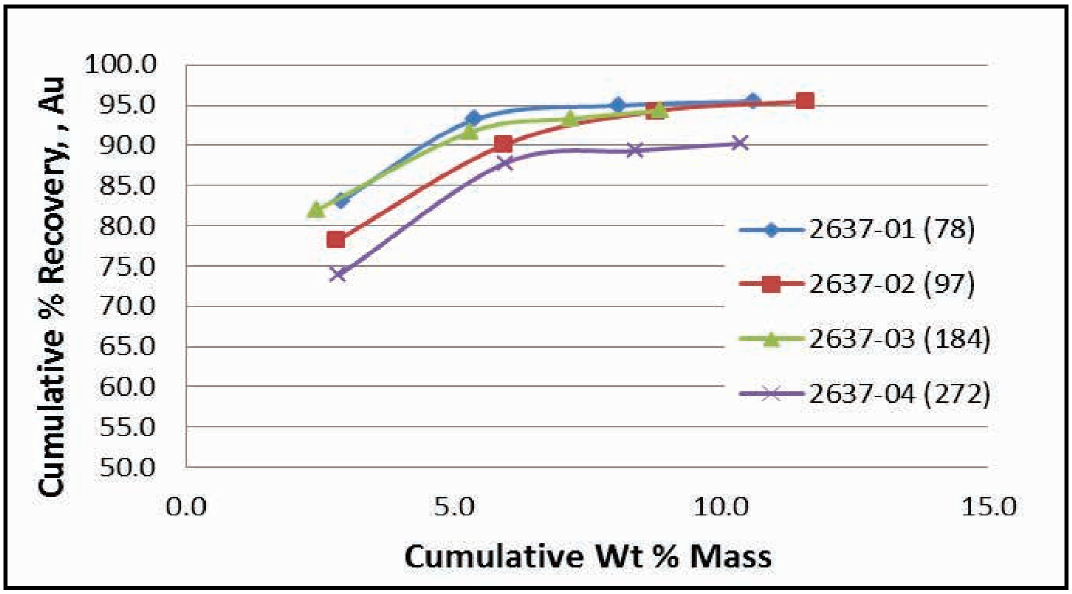



**Figure 13.5 Primary Grind Size versus Mass Recovery – G&T**




### PRIMARY FLOTATION REAGENTS

The flotation collector used was potassium amyl xanthate (PAX) which is a general-purpose flotation collector known for its ability to collect sulphide minerals efficiently. Methyl isobutyl carbinol (MIBC) was used as a frother to stabilize and collect the froth. The reagent CMC was used to depress the carbon; this is discussed in further detail in the following section. Reagent addition rates have not been completely optimized and increased rates of PAX were necessary in cases where higher TOC grades were present in the feed material tested. The dosage rates were used in the latest G&T test work has been used for design purposes but suitably adjusted to allow for scale-up from laboratory-sized tests to commercial-sized operations.

### Carbonaceous Material Rejection

A pre-flotation stage ahead of the rougher flotation circuit was tested, and was demonstrated to result in the reduction of the graphite content and the reagent consumption in the subsequent flotation stages. The pre-flotation stage has an impact that is evident in the cleaner circuit, resulting in the decrease in the amount of TOC available to report to this stage. Although initially masses of up to 10% were reporting to the pre-flotation concentrate stream, this was found to be dependent upon the amount of TOC present in the feed. Recent test work results indicate that a mass recovery of only about 5% will be obtained.

The use of the pre-flotation stage was anticipated to provide inconsistent results because of anticipated fluctuations in the TOC in the feed material; therefore further avenues for carbon rejection were explored. Also, although pre-flotation had a positive impact in reducing the carbon content, it was apparent from the test work results obtained that further intervention would be required to bring TOC levels to below 1%. Tests at SGS incorporating CMC as a depressant indicated that the use of this reagent would reduce the amount of carbon reporting to the concentrate material without detrimentally affecting gold recovery. The variable and often high gold content of the pre-flotation concentrate precluded the possibility of discarding this material.

G&T completed tests on a sample of argillite material with a higher-than-average feed TOC content, namely 1.25% TOC, from Composite Sample 871. A comparison of the results is shown in Table 13.15, and compared with the other results. The comparison includes the use of pre-flotation stage and the addition of CMC to the cleaner circuit, and then the rougher circuit for graphite depression. Although gravity recovery was incorporated during this test program, the results in this table have been re-calculated to record only the contribution of the flotation process to the gold recovery values.

### Table 13.15 Argillite Sample Flotation Parameter Study for Carbon Rejection – G&T

| Test No. | Primary Grind $P_{80}$, µm | Pre-flotation | CMC Addition Rougher/ Cleaner/ Cleaner (g/t) | PAX Rougher/ Cleaner (g/t) | Rougher Flotation Results | | | | Cleaner Flotation Results | | | | % Gold Reporting To | | | | Subsequent Leach Recovery Flotation Concentrate |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | Mass Recovery Weight (%) | Concentrate Grade Au (g/t) | % Recovery excluding Gravity | % TOC | Mass Recovery, Weight (%) | Concentrate Grade Au (g/t) | % Recovery Excluding Gravity | % TOC | Rougher Tailings | Pre-flotation Concentrate | 1st Cleaner Tailings | 2nd Cleaner Tailings | |
| KM2637-71 | 227 | Yes | 0/0/- | 90/30 | 10.08 | 5.90 | 91.49 | 1.59 | 6.60 | 8.95 | 90.96 | 1.91 | 6.14 | 2.37 | 0.53 | - | 80.4 |
| KM2637-89 | 227 | Yes | 0/50/15 | 90/30 | 11.39 | 7.07 | 86.60 | 1.30 | 4.92 | 15.38 | 81.50 | 0.28 | 10.40 | 3.01 | 1.35 | 3.78 | - |
| KM2637-92 | 164 | Yes | 0/50/15 | 90/40 | 11.77 | 6.67 | 91.35 | 1.39 | 4.70 | 16.00 | 88.14 | 0.28 | 6.24 | 2.41 | 2.41 | 0.80 | 95.7 |
| KM3185-1 | 164 | No | 0/50/15 | 90/40 | 10.96 | 6.03 | 86.23 | 2.38 | 3.12 | 20.66 | 84.10 | 0.31 | 13.77 | - | 1.14 | 0.99 | - |
| KM3185-2 | 164 | No | 0/50/15 | 120/40 | 12.54 | 4.64 | 81.31 | 2.31 | 2.86 | 19.22 | 76.78 | 0.31 | 18.69 | - | 3.60 | 0.94 | - |
| KM3185-3 | 164 | No | 0/50/15 | 150/40 | 13.61 | 5.88 | 91.04 | 2.46 | 2.92 | 25.65 | 85.12 | 0.36 | 8.96 | - | 5.27 | 0.65 | - |
| KM3185-4 | 164 | No | 120/50/15 | 90/40 | 9.69 | 7.57 | 91.14 | 1.24 | 4.23 | 17.06 | 89.68 | 0.32 | 8.86 | - | 0.81 | 0.64 | - |

Test 2637-71 is the base test which incorporated a pre-flotation stage but not any CMC addition. Test 2637-89 used pre-flotation followed by CMC addition to two stages of cleaning. The primary grind in both cases was slightly coarser than ideal, at a $P_{80}$ size of 227 µm. However, the TOC content of the cleaner flotation concentrate is significantly lower with Test 2637-89, although the gold loss to the intermediate tailings is still high at about 5%, namely the total of the first and second cleaner tailings.

Following Test 2637-89, Test 2637-92 incorporated a finer primary grind ($P_{80}$ of 164 µm), a pre-flotation stage and CMC addition. The rougher flotation results were comparable to the results of those of Test 2637-71. However, the cleaner flotation results were much improved in Test 2637-92, with a higher concentrate grade, lower concentrate mass and significantly lower TOC content. The overall recovery was slightly lower: 88% versus the 91% obtained for the base case Test 2637-71. However, the apparent reduction in flotation recovery is more than compensated for in the subsequent leach recovery of the concentrate at 95.7% gold recovery compared with 80.4% for Test 2637-71. The increased leach recovery is attributed to the lower TOC in the concentrate.

The positive results for the CMC addition to the cleaner circuit required that the rougher flotation circuit be investigated with respect to CMC addition to this stage, and the possible omission of the pre-flotation stage. The results of the G&T test program KM3185 indicated that when pre-flotation was not used in the rougher circuit and no CMC was used, the amount of collector reagent required to achieve acceptable rougher circuit recovery increased significantly. Test 3185-3 was comparable in rougher recovery to the results of Tests 2637-92 and 2637-71, although the mass of the rougher concentrate was higher, as was the TOC content in the rougher concentrate. The amount of gold lost to the rougher concentrate was equivalent to that of the rougher concentrate and the pre-flotation concentrate combined in Test 2637-92. However, with respect to the results obtained for Test 2637-92, there is the ability to recover some of the gold lost to the pre-flotation concentrate downstream, in the scavenger gravity concentration circuit recovery.

In Test 3185-3, the TOC was removed in the cleaner circuit to an acceptable level with the addition of CMC, but not to the same extent as where pre-flotation and the addition of CMC to the cleaners was incorporated.

The final test conducted, namely Test 3185-4, was designed to use CMC in the rougher and also the cleaner stages for carbon rejection. The amount of collector reagent was also reduced to the base test level. The results indicated that rougher recovery was good, with high TOC rejection at this stage and a reduced mass in the concentrate at an equivalent gold recovery. The results of this test are comparable to the tests which incorporated the pre-flotation stage. However, large amounts of reagent would be required for this scenario.

The use of CMC in the flotation circuit stages has not been completely optimized. The use of pre-flotation alone does not appear to be sufficiently robust to control the

TOC in the leach feed, while the use of CMC-only also suffers from potential operational oscillations in the results obtained should the TOC in the feed vary unexpectedly.

Although the addition of CMC improved the metallurgical performance of the flotation circuit, the pre-flotation stage will be retained to assist with the control of the carbonaceous material, should higher concentrations of carbon be present in the plant feed material from time to time.

#### CLEANER FLOTATION

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

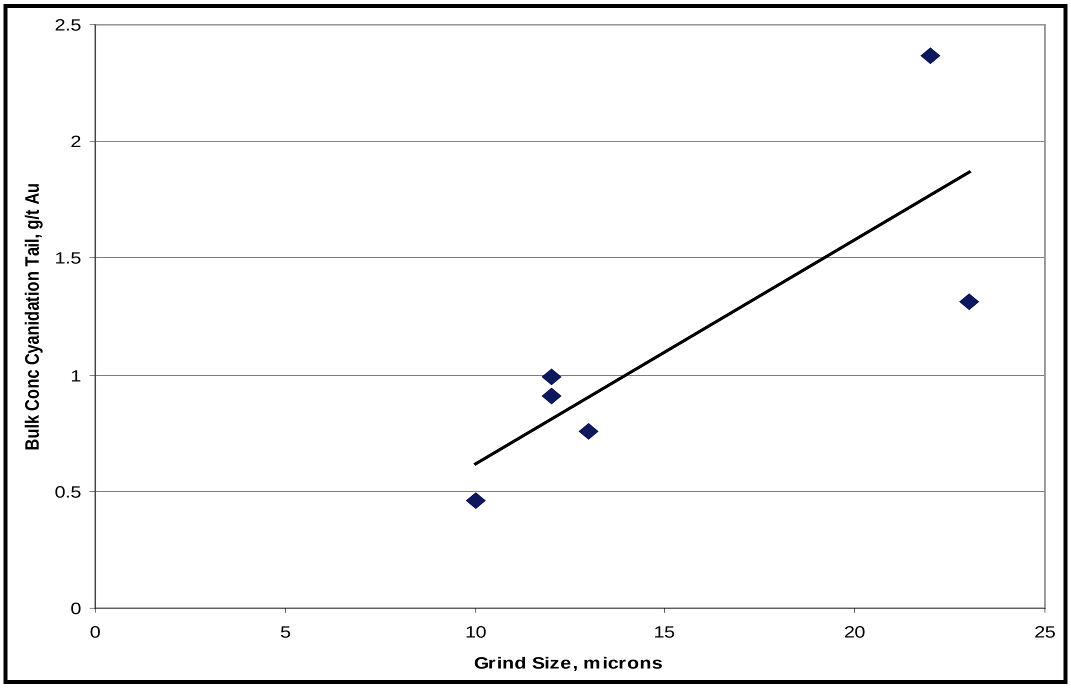
Previous test work indicated that incorporating more than one stage of cleaning had no positive impact, and possibly a negative impact, on the TOC content of the final concentrate; however, the addition of CMC decreased the TOC content of the final concentrate but required additional cleaning stages. Although the flotation circuit design is based on two stages of cleaning, it may be possible to simplify the circuit by using only one stage. This will be confirmed during the next phase of test work.

#### REGRIND OF CONCENTRATE

Test work has established that the extraction of gold from the flotation concentrate by cyanidation is sensitive to the fineness of grind. The 2010 PEA assumed a regrind size to $P_{80}$ size of 20 µm, which is the same value used in the current process design.

Figure 13.6 shows the gold content of flotation concentrate cyanidation tailings as a function of the regrind size of the concentrate. A finer regrind results in a lower tailings assay. Assuming a concentrate grade of 15 g/t gold, a 1 g/t decrease in the tailings assay represents a significant increase in the gold extraction and readily justifies a finer regrind size. There is some scatter around the results obtained and specific data cannot be extracted. However, the increasing gold dissolution with the increasing fineness of grind is apparent, and remains particularly the case for regrind sizes of $P_{80}$ less than 20 µm values. Additional test work to assess the potential of a finer regrind size $P_{80}$ than 20 µm will be undertaken.

**Figure 13.6 Flotation Concentrate Cyanidation Tailings versus Regrind Size – G&T**




### 13.2.8 CYANIDATION

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

- Whole-ore cyanidation at a primary grind $P_{80}$ of 74 µm results in gold extraction values of less than 10%.
- Direct cyanidation of flotation concentrates without a regrind prior to leaching resulted in low recoveries.
- The results from preg-robbing tests indicate that the samples tested displayed a natural tendency to have a very high preg-robbing activity.
- Subsequent test work and an analysis of the results obtained have indicated that low extraction values are as a result of both the presence of active carbon in the deposit, and the requirement of a fine regrind for improved gold liberation prior to leaching.

During the test work completed in 2010 and 2011, it became apparent that even with CIL leaching, control of the TOC content would be required in order to achieve acceptable leach recoveries of the flotation concentrate. The TOC content of the gravity concentrate is highly controlled by the method in which the gravity concentrator works, and is therefore not affected. The leaching of the two concentrate products will be discussed separately.

### Leaching of Gravity Concentrates

Initially, testing of the gravity concentrates included the use of CIL leaching for gold extraction. Cyanidation of the gravity concentrate consistently achieved a gold extraction greater than 97% as the nature of the gravity concentrate is that it leaches rapidly and it was also found that pre-aeration was not required. Gravity concentration, by nature of the mechanism of concentration, has an inherent ability to reject carbonaceous material. The resulting concentrate is a low mass, typically high grade, gold product. Although the gravity product is relatively coarse in nature, the gold is readily recoverable through the implementation of an intensive cyanidation circuit. Confirmatory tests were completed which indicate that without regrind prior to intensive cyanidation, a recovery of 92 to 97% can be realized from this product as shown by the test results in Table 13.16.

**Table 13.16 Gravity Concentrate, Cyanidation Test Results – G&T**

| Test No. | Composite | Regrind | Au Extraction (%) | NaCN Usage (kg/t) |
|---|---|---|---|---|
| 2637-97 | 872A | Yes | 97.8 | 9.2 |
| 2637-98 | 891 | Yes | 98.1 | 8.8 |
| 2637-99 | 894 | Yes | 98.6 | 6.9 |
| 2637-100 | 871 | Yes | 97.0 | 6.9 |
| 3185-5 | 1 to 4 | No | 97.3 | 16.4 |
| 3185-6 | 1 to 4 | Yes | 96.8 | 20.2 |
| 3185-21 | 14/16/17 | No | 92.0 | 6.2 |

In the design of the plant, following the intensive cyanidation of the gravity concentrate, the residue material will be combined with the flotation concentrate for regrinding and CIL leaching. The overall gold extraction from the gravity concentrate using this process flowsheet is projected at 98% of which about 90% will be leached during the intensive cyanidation stage, and the balance will be recovered following regrinding and additional leaching.

### CIL Leaching of Flotation Concentrates

During the test work undertaken, the regrinding of the flotation concentrates prior to leaching was indicated to be essential for high gold dissolution values to be obtained. Although the flotation concentrate appears to benefit from a regrind size as fine as 10 µm, further investigation is required to confirm the optimal regrind size as previously discussed.

G&T conducted several leach tests as part of the KM2637 test program. Initially, leach tests were done for only a period of 24 hours. The results from a number of tests indicated that the material required both further regrinding of the concentrate, and additional cyanide, as well as longer leach duration periods, in order to achieve extractions of between 50 and 89%. A summary of the test results is shown in Table 13.17.

**Table 13.17 Cyanidation Test Work Results – G&T**

| Composite | Leach Test No. | Leach Feed Source | Leach Feed Grade Au (g/t) | % Leach Extraction | % TOC |
|---|---|---|---|---|---|
| 1 | 19 | Bulk concentrate | 7.6 | 81.0 | - |
| 2 | 20 | Bulk concentrate | 8.4 | 44.6 | - |
| 3 | 21 | Bulk concentrate | 10.2 | 83.6 | - |
| 1 | 22 | Test 19 tailings | - | 90.0 | 1.29 |
| 2 | 23 | Test 20 tailings | - | 49.0 | 3.46 |
| 3 | 24 | Test 21 tailings | - | 89.0 | 1.09 |
| 3 | 27 | Bulk concentrate | 7.29 | 92.3 | 1.22 |
| 2 | 26 | Bulk concentrate | 3.78 | 58.4 | 2.80 |
| 1 | 34 | Bulk concentrate | 3.34 | 90.3 | 0.97 |
| 3 | 28 | Gravity concentrate | 26.8 | 97.9 | 0.18 |
| 2 | 32 | Gravity concentrate | 25.0 | 71.4 | 0.48 |
| 1 | 33 | Gravity concentrate | 8.97 | 86.0 | 0.16 |

Leach Tests 22 through 24 consisted of a re-leach of the residues from Tests 19 through 21, resulting in a modest increase in overall gold extraction. Prior to this re-leaching step, the residues were milled to a $P_{80}$ size of 9 µm. Of the three composites tested, Composite 2 reported the lowest gold extraction, and also had the highest graphite content.

Tests 26 through 28 included regrinding of the concentrates and pre-aeration prior to cyanidation. Even under these conditions, the Composite 2 material reported lower gold extraction. The variation in gold extraction from these tests appears to correlate with the variation in graphite content and the impact of this content is not lessened by pre-aeration, nor by using the CIL test procedure.

As part of the development test work conducted by SGS, the flotation concentrates from 16 composite samples were subjected to cyanidation tests. The concentrate slurry was pre-aerated for a variable time before being leached under CIL test procedure conditions for 48 hours. The test results obtained are summarized in Table 13.18.

**Table 13.18 Flotation Concentrate Cyanidation Results – SGS**

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

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

G&T subjected Composite 4 material to a number of additional tests in order to assess the flotation variables and leach properties of the product created. The results obtained in testing the parameters which may have affected the leach recovery are shown in Table 13.19.

**Table 13.19 Flotation Concentrate Variables – G&T**

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

A gold leach extraction of between 78.9 and 91.5% was achieved in the tests. The primary grind size and pre-flotation time did not seem to influence gold recovery. The pre-flotation stage did lower the TOC content, in the concentrate, although the content of the TOC recovered into the concentrate appears to have a detrimental effect on leach recovery. The regrind $P_{80}$ size also appears to have influenced gold extraction.

The results of additional leach test work using Composite 4 material are provided in Table 13.20. The feed material for these leach tests was created from a sample which had been gravity processed, had included the pre-flotation stage, and then had CMC added to the cleaner flotation stages, except for Test 74 which had a very low TOC content prior to processing and did not have CMC added to the cleaner circuit.

**Table 13.20 Composite 4 Variability Samples - G&T**

| CIL Test No. | Sample No. | Regrind $P_{80}$ µm | Feed % TOC | Flotation Concentrate % TOC | % Au Extraction CIL |
|---|---|---|---|---|---|
| 93 | 872a | 18 | 1.13 | 0.31 | 92.7 |
| 94 | 891 | 19 | 1.23 | 0.26 | 93.9 |
| 95 | 894 | 16 | 0.80 | 0.23 | 97.4 |
| 96 | 871 | 17 | 1.25 | 0.28 | 95.5 |
| 74 | 865-3 | 11 | 0.21 | 0.11 | 96.3 |
| - | 872b | Not completed | | - | - |

The results obtained indicate that the average recovery from the flotation concentrate using the CIL-test procedure was 95%. While 24 hours appears to be an adequate leach time for non-argillite materials, for the materials with significant TOC content, a leach time of 48 hours appears warranted even after implementation of pre-flotation and CMC addition to the cleaner circuit.

With graphite removal using a pre-flotation stage, and with graphite depression using CMC during cleaner flotation, an average gold extraction by cyanidation of 95% was achieved in the G&T test work, and 94.5% in the SGS test work.

### 13.2.9 Variability Test Work

SGS commenced a variability test work program in 2010, but this developed it into a method to investigate the potential of de-sliming as an alternative to the pre-flotation for graphite removal, as well as to evaluate the impact of graphite depressants on TOC reduction during the cleaner flotation stages. Although both techniques were successful, the lack of definition of the de-sliming process rendered this method unusable, and the CMC addition method was pursued as previously discussed.

Regarding the variability test results, SGS also reported a wide range of gold recoveries and concentrate grades prior to cyanidation. However, typically since the gravity concentration results are not reported separately from the flotation recovery results, an overall flotation effect cannot be evaluated. The overall recovery after cyanidation ranged from 44 to 85% gold recovery. As mentioned, the test procedure used de-sliming without adequately detailing the procedure, as well as using three stages of flotation concentrate cleaning to produce a flotation concentrate which is not consistent with the chosen flowsheet design. For this reason, these results obtained by SGS in this test program were not reviewed.

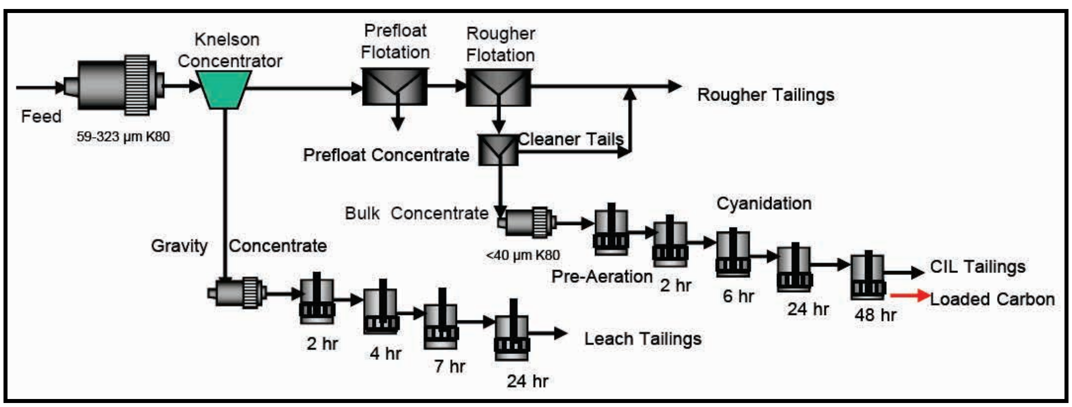
Variability test work was also carried out at G&T on samples taken from the deposit, to provide data for process design criteria, and to test the variability among the samples using the elected processing method. The composite samples included in this test work were from various drillholes located across the deposit.

The variability test procedure was based on the flowsheet and test conditions developed during all the test programs. The test conditions and test procedure used at G&T are depicted in Table 13.21 and Figure 13.7.

**Table 13.21 Variability Flotation Test Conditions – G&T**

| | Reagents Added (g/t) | | | Time (minutes) | | | |
|---|---|---|---|---|---|---|---|
| **Flotation Stage** | **CMC** | **PAX** | **MIBC** | **Grind** | **Condition** | **Float** | **pH** |
| Feed/Natural | - | - | - | - | - | - | 8.5 |
| Pre-flotation | - | - | 30 | - | 1 | 8 | 8.5 |
| **Bulk Circuit** | | | | | | | |
| Rougher 1 | 50 | 30 | 30 | - | 1 | 2 | 8.5 |
| Rougher 2 | 50 | 30 | 15 | - | 1 | 2 | 8.5 |
| Rougher 3 | 20 | 30 | 8 | - | 1 | 4 | 8.6 |
| Cleaner 1 | 50 | 30 | 8 | - | 1 | 8 | 8.5 |
| Cleaner 2 | 15 | 10 | 4 | - | 1 | 6 | 8.8 |

**Figure 13.7 Variability Test Procedure for Gravity, Flotation and Leaching – G&T**



In each case, the composite sample was prepared from a contiguous series of drillhole samples. The results of the variability tests using the composite samples are summarized in Table 13.22.

The overall recovery values shown in Table 13.22 include the gravity concentrate and the bulk cleaner flotation concentrate together with an assumed 40% of the pre-flotation plus cleaner tailings. In some instances, a variability test was repeated. Test 10 on material from Composite 9 was carried out at a very coarse grind with a resultant significant loss in gold recovery. Test 36—which was identical to Test 10, but at the correct normal grind size—achieved the expected recovery. The target primary grind $P_{80}$ was 184 µm and the actual $P_{80}$ grind sizes obtained during testing, are shown in Table 13.22.

**Table 13.22 Variability Tests, Gravity and Flotation Results – G&T**

| Test No. | Composite Sample Identification | Primary Grind $P_{80}$ µm | Head Assays | | | Tailings Assays | Gravity+Flotation, % Recovery to Concentrate |
|---|---|---|---|---|---|---|---|
| | | | Au (g/t) | S (%) | TOC (%) | Au (g/t) | |
| 10 | 9 | 323 | 0.53 | 1.07 | 0.92 | 0.06 | 86.1 |
| 36 | 9 | 197 | 0.49 | 1.44 | 1.18 | 0.02 | 95.3 |
| 11 | 10 | 194 | 0.42 | 2.09 | 1.37 | 0.04 | 81.4 |
| 12 | 18 | 145 | 0.48 | 4.14 | 1.53 | 0.11 | 77.8 |
| 27 | 18 | 134 | 0.40 | 3.95 | 1.51 | 0.05 | 82.8 |
| 13 | 19 | 171 | 1.68 | 3.70 | 1.55 | 0.08 | 94.1 |
| 14 | 3 | 169 | 0.32 | 0.86 | 0.15 | 0.02 | 89.9 |
| 15 | 6 | 184 | 1.22 | 2.32 | 1.09 | 0.04 | 94.8 |
| 16 | 7 | 148 | 0.28 | 0.71 | 0.11 | 0.02 | 89.9 |
| 17 | 8 | 181 | 0.38 | 0.55 | 0.07 | 0.01 | 92.1 |
| 18 | 20 | 139 | 1.25 | 3.64 | 1.68 | 0.06 | 94.5 |
| 29 | 928A | 147 | 0.44 | 3.18 | 1.22 | 0.04 | 90.3 |
| 30 | 928B | 175 | 0.37 | 3.14 | 1.79 | 0.03 | 89.8 |
| 31 | 928C | 140 | 0.78 | 2.43 | 0.70 | 0.03 | 94.7 |
| 32 | 928D | 144 | 0.44 | 2.15 | 1.66 | 0.03 | 93.0 |
| 33 | 939A | 134 | 0.73 | 3.90 | 2.03 | 0.10 | 85.7 |
| 34 | 939B | 131 | 0.26 | 4.55 | 1.81 | 0.15 | 43.6 |
| 35 | 981 | 125 | 0.88 | 1.23 | 0.36 | 0.04 | 92.7 |

Figure 13.8 and Figure 13.9 (which are based on Table 13.22) indicate that, under the conditions tested, both the concentration of sulphur and the TOC in the feed influence the gold lost in the rougher flotation tailings. A detailed analysis of the results in Table 13.22 indicated that this particular test program was not completed with sufficient confidence regarding assaying, sampling of products, and processing techniques; therefore these effects should only be used as a guide at this stage of the Project.

**Figure 13.8 Variability Tests, Tailings Gold Grade versus Feed Sulphur Grade – G&T**

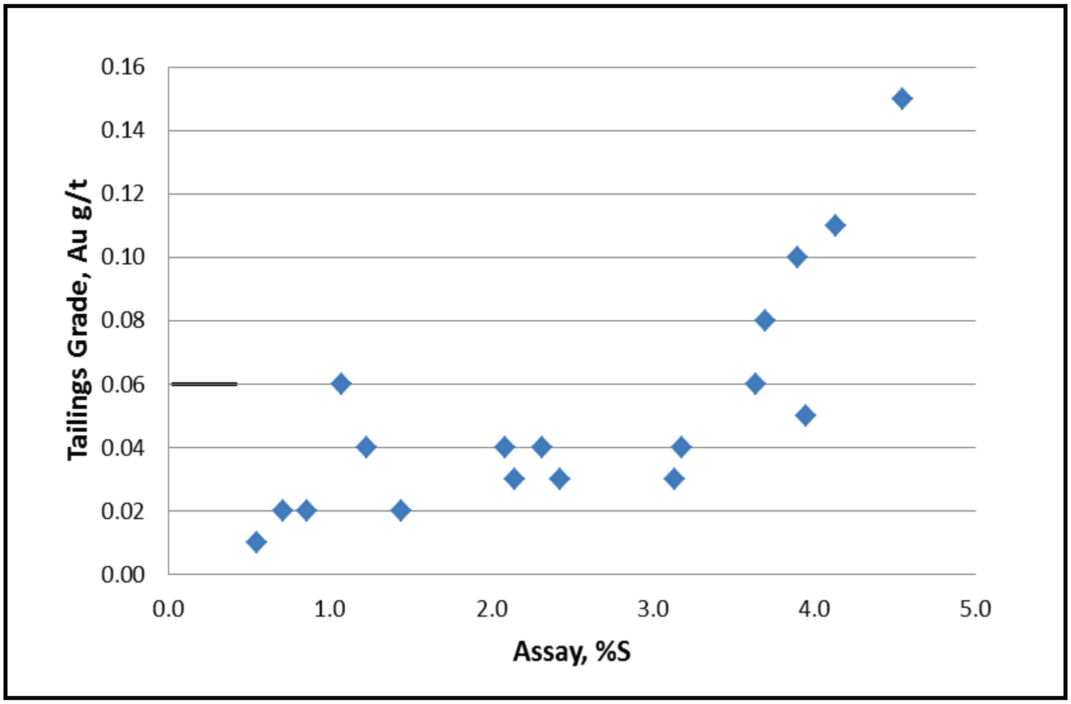
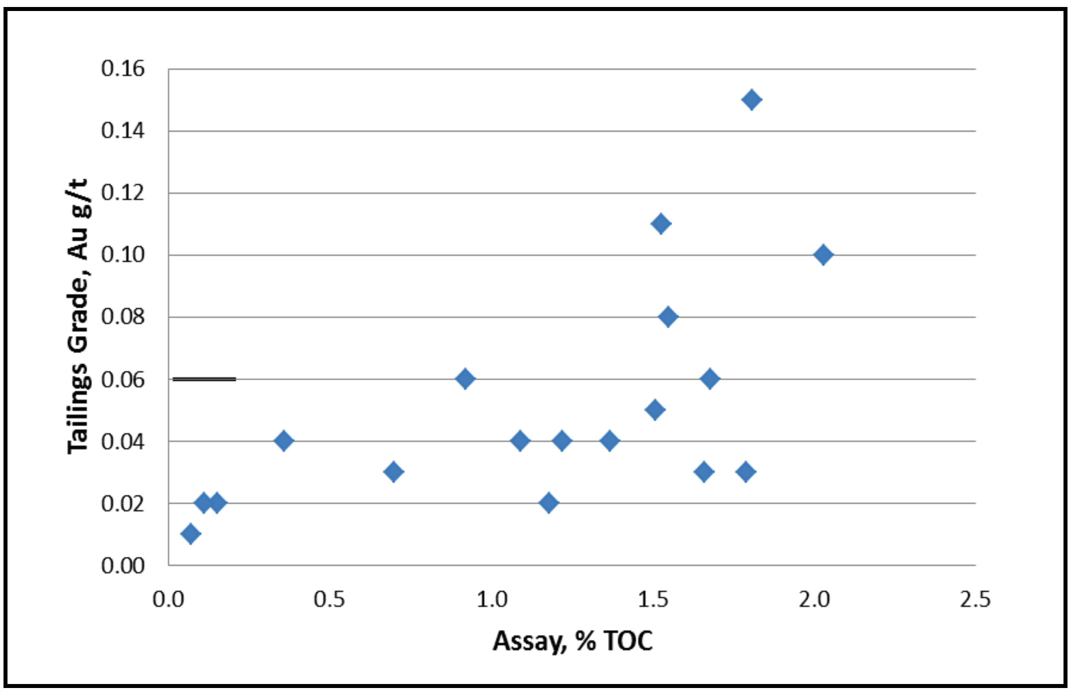



**Figure 13.9 Variability Tests, Tailings Gold Grade versus Feed TOC Grade – G&T**



As shown by Figure 13.8 and Figure 13.9, an increase in the feed TOC value and the sulphur content of the feed increases the gold loss to the tailings. However, there are a number of processing steps which could influence the behaviour of the sulphur and the TOC and ultimately their effect on gold content of the tailings. These processing steps could be inter-related to varying degrees. Tetra Tech therefore recommends that the potential interaction of a number of processing conditions and variables be established. These conditions and variables include, but are not limited to, investigating the effects of the following:

- the effect of primary grind size
- the effects of dosage rates of both PAX and CMC, and PAX plus CMC, to determine whether synergies exist
- the overall influence of the pre-flotation stage
- verification of the sampling procedure adopted during the test work
- verification of the accuracy of the assaying techniques used
- determination of the influence of the mass recovery of the rougher stage, and the subsequent cleaner steps.

These effects and influences should be established for the flotation process testing only, exclusive of the gravity concentration step.

If the effects shown in Figure 13.8 and Figure 13.9 are confirmed, the mine plan will need to take into consideration the sulphur content and TOC in the plant feed material as a more detailed plan is developed.

### 13.2.10 SUMMARY

Based on the test work results obtained using the flowsheet which has been adopted for the treatment of the gold-bearing SMG material, the overall gold recovery incorporating all the various processes is anticipated to be just over 90%. For purposes of the design, the calculated overall gold recovery is 90.3%, made up of 21.6% recovery after intensive cyanidation from the gravity circuit, and 68.7% recovery from the flotation, secondary gravity concentration and subsequent CIL and smelting processes.

# 14.0 MINERAL RESOURCE ESTIMATES

The following section is derived directly from the report "Technical Report on an Updated Mineral Resource Estimate on the Spanish Mountain Gold Deposit", dated August 31, 2012. Minor changes have been made for report consistency.

At the request of Brian Groves, President and CEO of SMG, Giroux Consultants was retained to produce a resource estimate on the Project located approximately 6 km east of Likely, BC and 70 km northeast of Williams Lake. The effective date for this resource estimate is June 18, 2012.

G.H. Giroux is the QP responsible for the resource estimate. Mr. Giroux is a QP by virtue of education, experience and membership in a professional association. He is independent of both the issuer and the vendor applying all of the tests in Section 1.5 of NI 43-101. Mr. Giroux visited the Property on June 29, 2011.

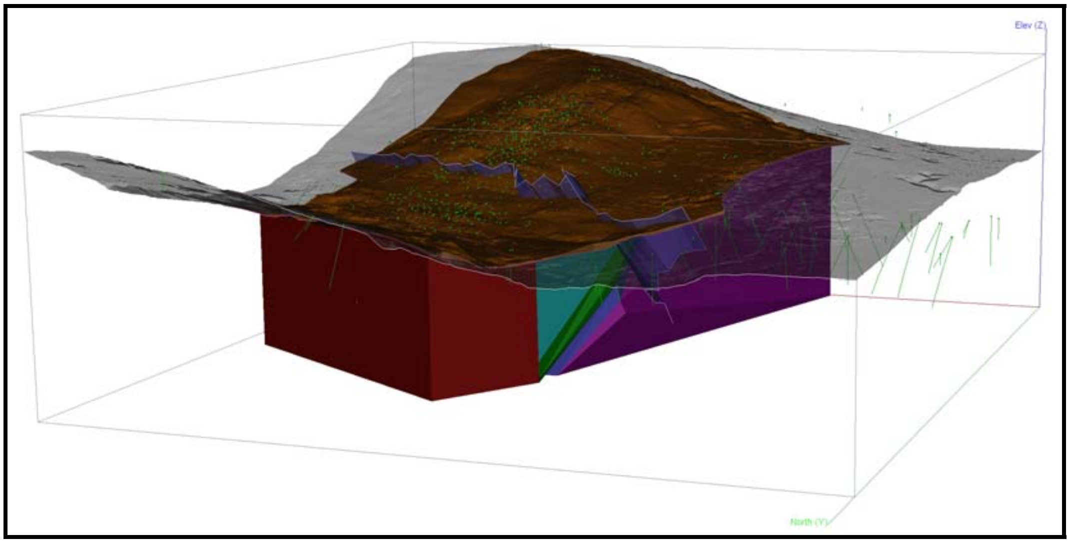
## 14.1 DATA ANALYSIS

In total, 816 drillholes were provided, but only 670 diamond and 71 RC drillholes penetrated the various geologic solids (see Appendix B for the drillhole list). This update is based on an additional 142 infill diamond drillholes completed since the previous 2011 mineral resource estimate (Giroux and Koffyberg 2011). Missing or unsampled intervals were filled with 0.001 g/t gold. Samples not sampled for silver from earlier drill campaigns were left blank as were samples not sampled for calcium, sulphur or arsenic. The assays statistics are shown below.

A 3D geologic model was produced by SMG geologist Alex Gow using Vulcan 3D mining software. As shown in Figure 14.1 and Figure 14.2, the Main Zone mineralization was modelled into an Upper Argillite unit, an Altered Siltstone unit, a Tuff unit and a Lower Argillite unit. The North Zone Argillite was modelled as a separate solid.

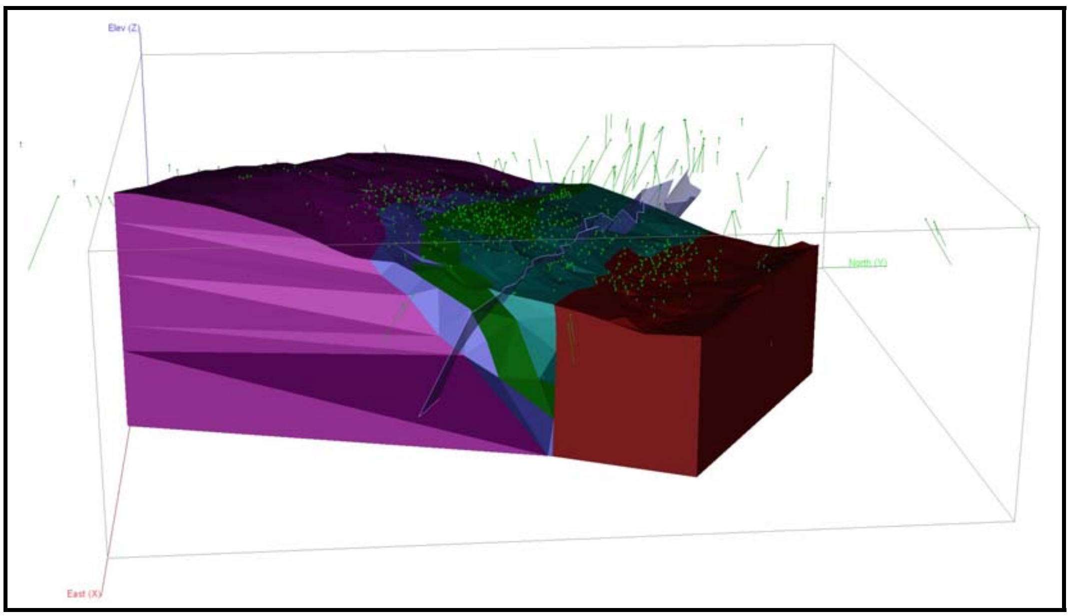
All material, outside of these domains, was considered waste.

**Figure 14.1 Isometric View Looking Southeast**




Note: Lower Argillite shown in purple, tuff in blue, Upper Argillite in green, siltstone in blue green and North Zone Argillite in red. Inflection plane shown in blue, surface topography in grey and overburden in brown.

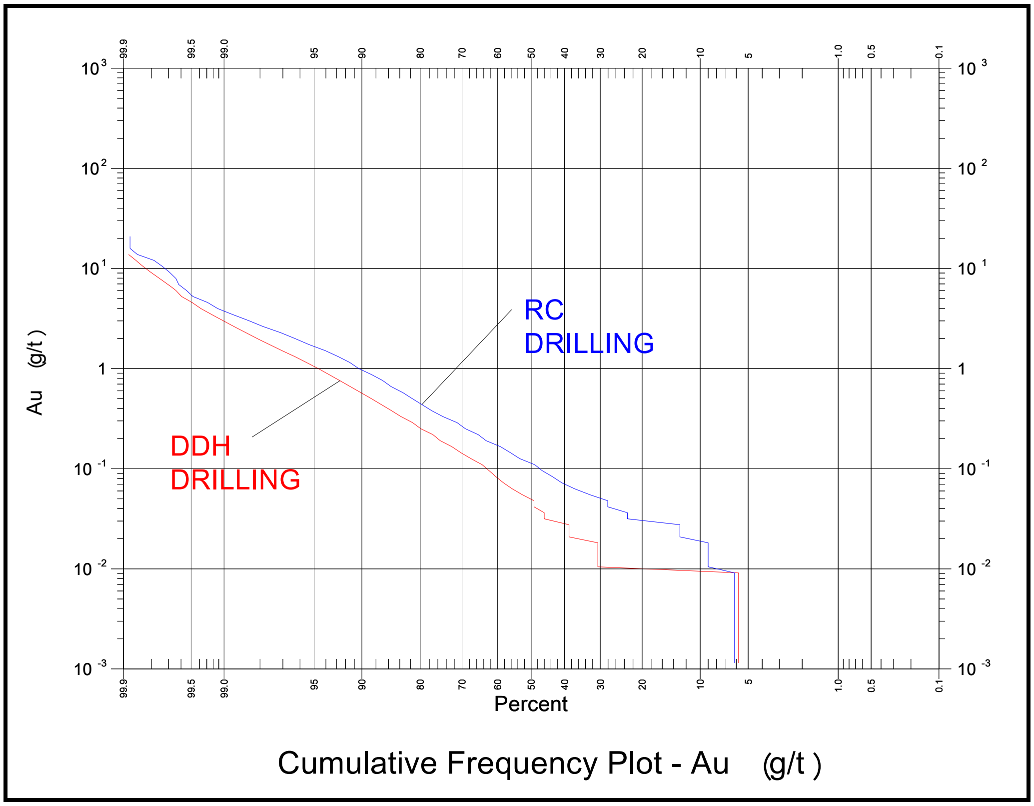
**Figure 14.2 Isometric View Looking West**




Note: Lower Argillite shown in purple, tuff in blue, Upper Argillite in green, siltstone in blue green and North Zone Argillite in red. Inflection plane shown in blue.

During the 2004 and 2006 drill campaigns, 113 RC drillholes were completed on the Property. Of these 71 intersected the mineralized domains. To determine if the RC results should be used in the resource estimate the grade distribution for gold from RC drilling was compared to the gold distribution from diamond drillholes, within the same volume of rock. The grade distributions are shown below in Figure 14.3. The RC drill results show a fixed bias with grades higher in all percentiles. Based on this bias and the fact the RC drillholes are all in areas tested by diamond drillholes, the RC results were not used in the resource estimate.

**Figure 14.3 Lognormal Cumulative Frequency Plot for Gold**




Note Diamond drilling (red) and gold from reverse circulation drilling (blue).

The sample statistics for gold and silver are tabulated in Table 14.1 and Table 14.2, subdivided by the various geologic domains. Only assays from diamond drillholes are used.

**Table 14.1 Statistics for Diamond Drillhole Gold Assays in Geologic Domains**

| | Upper Argillite | Altered Siltstone | Tuff | Lower Argillite | North Zone Argillite | Waste |
|---|---|---|---|---|---|---|
| Number of Assays | 13,274 | 9,508 | 19,902 | 34,291 | 16,192 | 5,478 |
| Mean Gold (g/t) | 0.436 | 0.070 | 0.320 | 0.208 | 0.242 | 0.060 |
| Standard Deviation | 1.397 | 0.786 | 2.715 | 1.82 | 0.801 | 1.003 |
| Minimum Value | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 |
| Maximum Value | 83.40 | 39.00 | 225.00 | 241.00 | 54.40 | 73.80 |
| Coefficient of Variation | 3.21 | 11.26 | 8.49 | 8.75 | 3.35 | 16.78 |

**Table 14.2 Statistics for Diamond Drillhole Silver Assays in Geologic Domains**

| | Upper Argillite | Altered Siltstone | Tuff | Lower Argillite | North Zone Argillite | Waste |
|---|---|---|---|---|---|---|
| Number of Assays | 13,124 | 9,272 | 19,253 | 33,510 | 15,962 | 5,475 |
| Mean Silver (g/t) | 0.87 | 0.40 | 0.44 | 0.59 | 0.66 | 0.57 |
| Standard Deviation | 1.28 | 0.66 | 1.20 | 0.76 | 1.40 | 1.00 |
| Minimum Value | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 |
| Maximum Value | 88.90 | 28.20 | 84.10 | 30.00 | 103.00 | 23.00 |
| Coefficient of Variation | 1.48 | 1.67 | 2.74 | 1.29 | 2.12 | 1.73 |

The gold grade distributions within the mineralized domains were examined to determine if capping was required and if so at what level. In each case the distribution for gold was strongly skewed. A lognormal cumulative frequency plot was produced for gold in each domain and in all cases showed multiple overlapping lognormal populations. As shown in Table 14.3, capping levels were determined to reduce the effect of small high grade populations that can be considered erratic. A similar procedure was used to cap silver values.

**Table 14.3 Capping Levels for Gold and Silver Assays in Geologic Domains**

| Domain | Cap Level Au (g/t) | Number Capped | Cap Level Ag (g/t) | Number Capped |
|---|---|---|---|---|
| Upper Argillite | 13.0 | 12 | 20.0 | 4 |
| Tuff | 30.0 | 15 | 30.0 | 4 |
| Altered Siltstone | 10.0 | 9 | 20.0 | 2 |
| Lower Argillite | 16.0 | 20 | 25.0 | 3 |
| North Zone Argillites | 15.0 | 5 | 30.0 | 5 |
| Waste | 2.0 | 3 | 10.0 | 2 |

The results from capping are shown in Table 14.4.

**Table 14.4 Statistics for Capped Gold and Silver Assays in Geologic Domains**

| | Upper Argillite | Altered Siltstone | Tuff | Lower Argillite | North Zone Argillite | Waste |
|---|---|---|---|---|---|---|
| **Capped Gold Assays** | | | | | | |
| Number of Assays | 13,274 | 9,508 | 19,902 | 34,291 | 16,192 | 5,478 |
| Mean Gold (g/t) | 0.418 | 0.058 | 0.289 | 0.189 | 0.237 | 0.047 |
| Standard Deviation | 0.844 | 0.402 | 1.404 | 0.674 | 0.584 | 0.111 |
| Minimum Value | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 |
| Maximum Value | 13.00 | 10.00 | 30.00 | 16.00 | 15.00 | 2.00 |
| Coefficient of Variation | 2.02 | 6.91 | 4.85 | 3.56 | 2.46 | 2.39 |
| **Capped Silver Assays** | | | | | | |
| Number of Assays | 13,124 | 9,272 | 19,253 | 33,510 | 15,962 | 5,475 |
| Mean Silver (g/t) | 0.86 | 0.40 | 0.43 | 0.59 | 0.65 | 0.57 |
| Standard Deviation | 0.99 | 0.61 | 0.95 | 0.75 | 1.04 | 0.95 |
| Minimum Value | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 |
| Maximum Value | 20.00 | 20.00 | 30.00 | 25.00 | 30.00 | 10.00 |
| Coefficient of Variation | 1.16 | 1.54 | 2.25 | 1.28 | 1.60 | 1.66 |

All domains were combined to determine the statistics for calcium, sulphur and arsenic.

**Table 14.5 Statistics for Calcium, Sulphur and Arsenic in all Domains**

| | Calcium (%) | Sulphur (%) | Arsenic (ppm) |
|---|---|---|---|
| Number of Assays | 98,412 | 38,638 | 98,598 |
| Mean Calcium, Sulphur, Arsenic | 3.09 | 1.34 | 68.9 |
| Standard Deviation | 1.35 | 1.33 | 72.4 |
| Minimum Value | 0.01 | 0.01 | 1.0 |
| Maximum Value | 12.50 | 10.00 | 2,680.0 |
| Coefficient of Variation | 0.44 | 0.99 | 1.05 |

No calcium, sulphur or arsenic assays required capping.

## 14.2 Composites

The drillholes were "passed through" the mineralized solids with the point at which each drillhole entered and left the solid recorded. Uniform 2.5 m down hole composites were then produced to honour these mineralized boundaries. Intervals less than 1.25 m at the solid boundaries were combined with adjoining intervals to produce a uniform support of 2.5, ±1.25 m. The statistics for 2.5 m composites are shown in Table 14.6 and Table 14.7.

**Table 14.6 Statistics for 2.5 m Gold and Silver Composites**

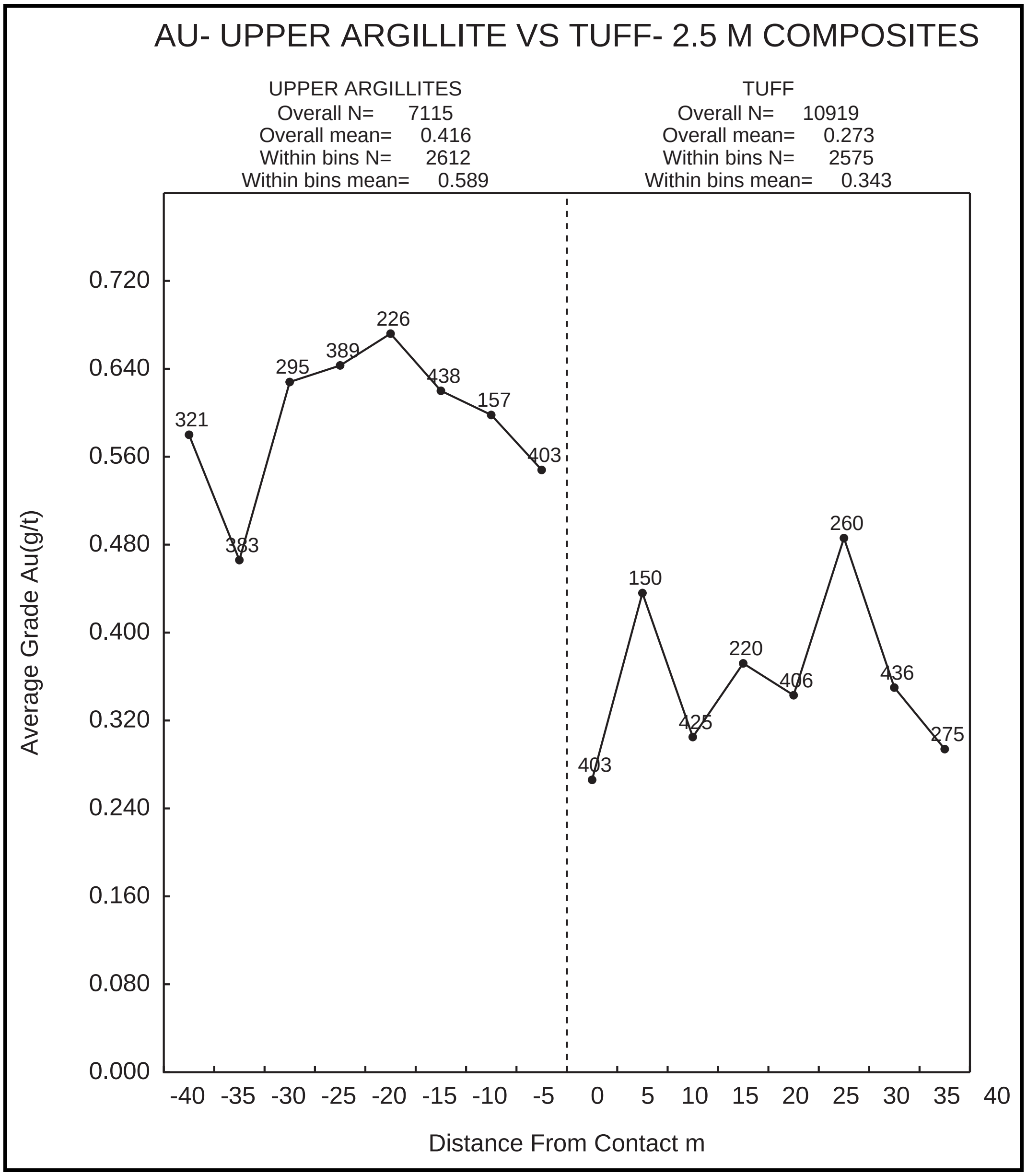
| | Upper Argillite | Altered Siltstone | Tuff | Lower Argillite | North Zone Argillite | Waste |
|---|---|---|---|---|---|---|
| **2.5 m Gold Composites** | | | | | | |
| Number of Composites | 8,295 | 5,921 | 12,200 | 21,509 | 10,396 | 3,373 |
| Mean Gold (g/t) | 0.409 | 0.055 | 0.267 | 0.183 | 0.233 | 0.048 |
| Standard Deviation | 0.682 | 0.255 | 0.879 | 0.492 | 0.415 | 0.093 |
| Minimum Value | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 |
| Maximum Value | 12.34 | 6.16 | 24.58 | 14.72 | 9.03 | 1.29 |
| Coefficient of Variation | 1.67 | 4.63 | 3.29 | 2.69 | 1.79 | 1.96 |
| **2.5 m Silver Composites** | | | | | | |
| Number of Composites | 8,261 | 5,881 | 12,054 | 21,231 | 10,354 | 3,373 |
| Mean Silver (g/t) | 0.86 | 0.40 | 0.42 | 0.59 | 0.65 | 0.59 |
| Standard Deviation | 0.93 | 0.50 | 0.69 | 0.63 | 0.83 | 0.89 |
| Minimum Value | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 |
| Maximum Value | 20.00 | 12.4 | 26.14 | 13.74 | 23.39 | 8.44 |
| Coefficient of Variation | 1.08 | 1.26 | 1.64 | 1.07 | 1.28 | 1.51 |

**Table 14.7 Statistics for Calcium and Sulphur and Arsenic within 2.5 m Composites from all Domains**

| | Calcium (%) | Sulphur (%) | Arsenic (ppm) |
|---|---|---|---|
| Number of Composites | 58,032 | 24,285 | 58,104 |
| Mean Calcium, Sulphur, Arsenic | 3.05 | 1.37 | 71.2 |
| Standard Deviation | 1.21 | 1.27 | 66.9 |
| Minimum Value | 0.01 | 0.01 | 1.0 |
| Maximum Value | 11.35 | 8.83 | 2,218.2 |
| Coefficient of Variation | 0.40 | 0.93 | 0.94 |

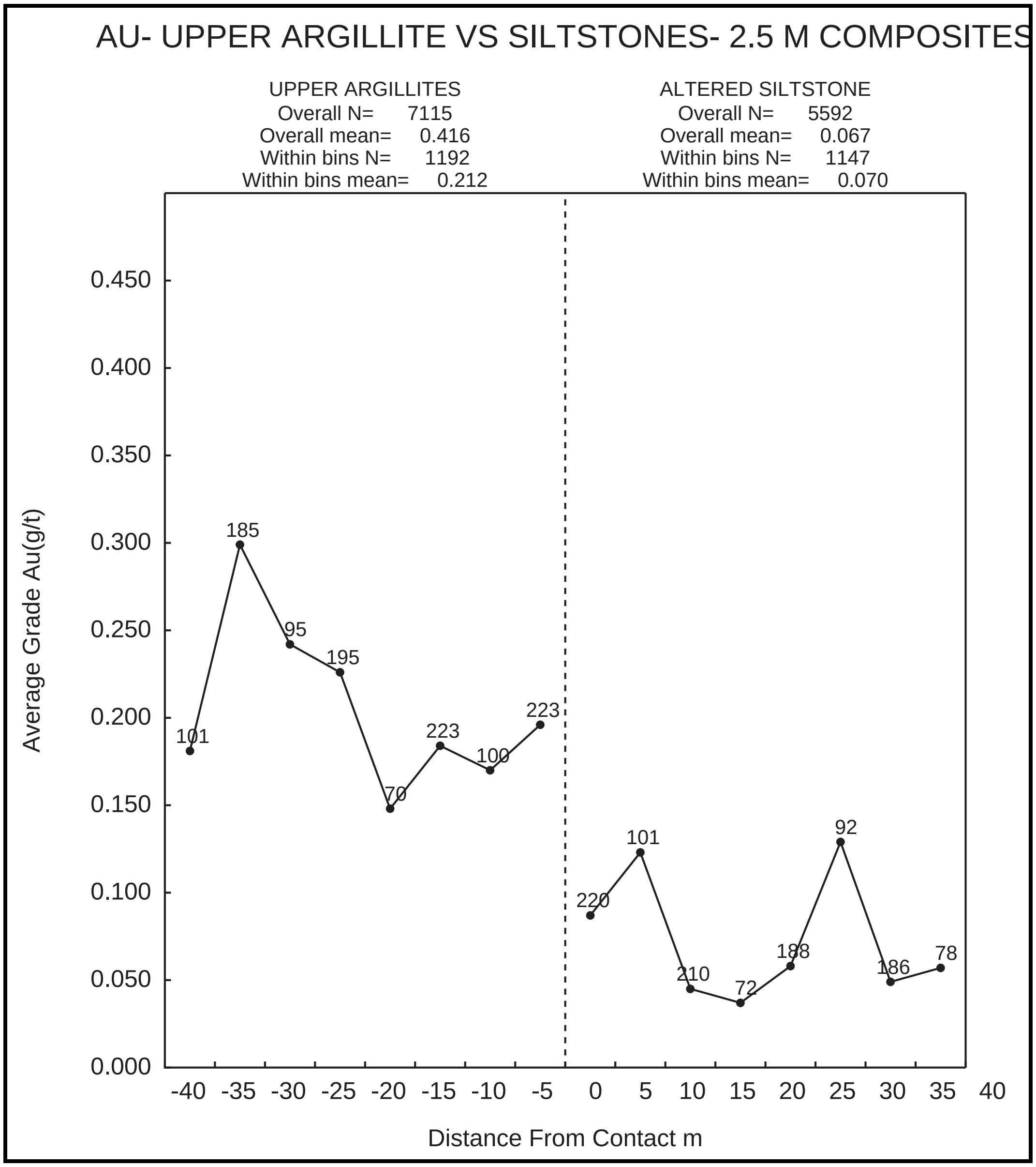
The gold grade relationships among the various lithologies across contacts were explored using contact plots. Figure 14.4 shows a contact plot for gold in the Upper Argillite compared with the Tuff domain. The dashed vertical line represents the contact between these two units and the average grade for gold is show on both sides for samples extending away from this contact. It is clear that there is a sharp grade change going across this contact and as a result, there should be a hard boundary for grade estimation. A hard boundary means samples on one side are not used to estimate blocks on the other side.

**Figure 14.4 Contact Plot for Gold in Upper Argillite versus Tuff Domain**



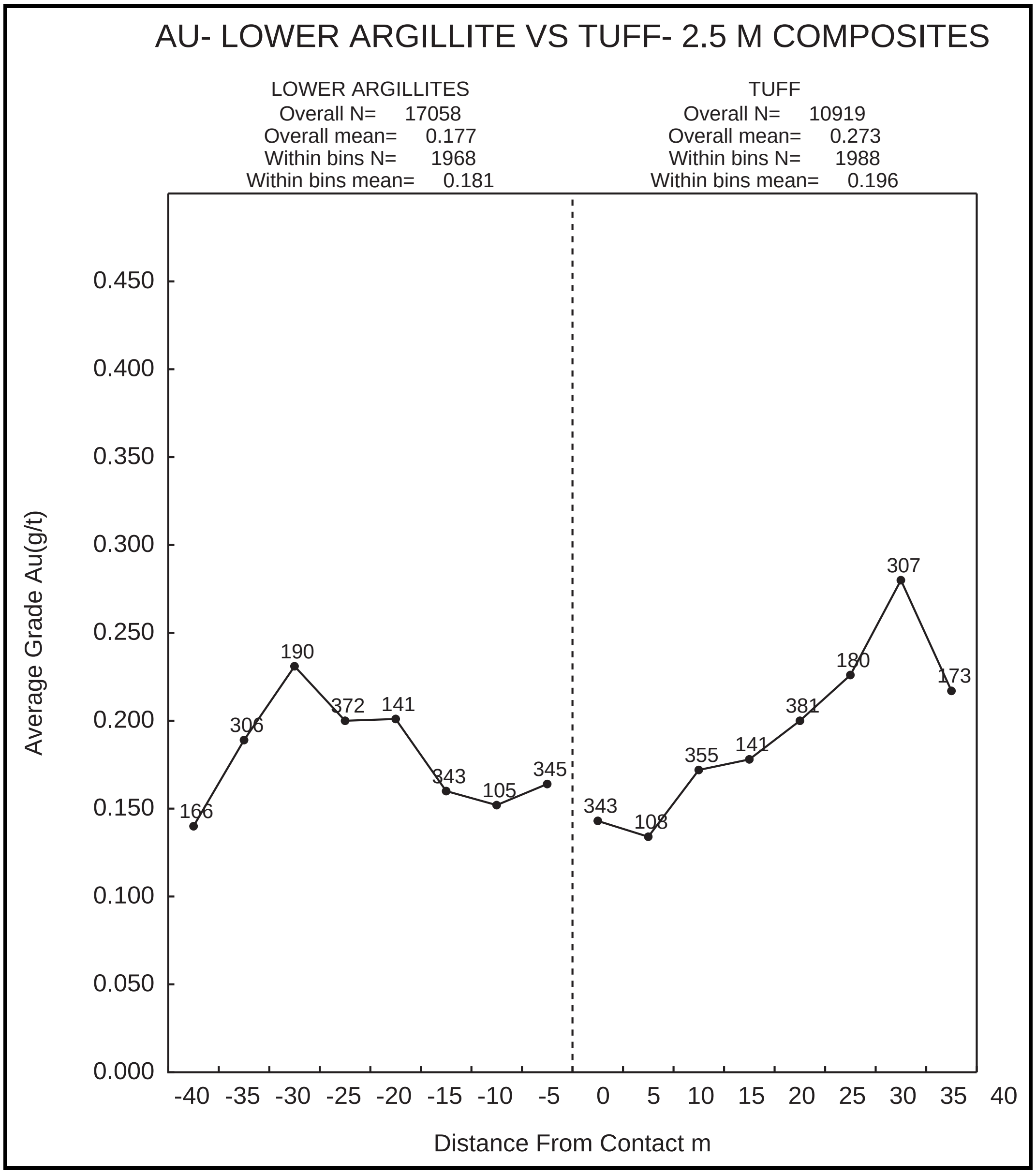

A similar plot for Upper Argillite and Altered Siltstone shows a similar sharp contact across the contact (Figure 14.5) and again a hard boundary should be imposed for grade estimation.

**Figure 14.5 Contact Plot for Gold in Upper Argillite versus Altered Siltstone Domain**




A contact plot for gold between the Lower Argillite and Tuff domain showed no significant changes across the contact (Figure 14.6) and these domains could be estimated with a soft boundary meaning composites from either could be used during estimation of blocks near this contact. The Upper and Lower Argillites and the Lower Argillite and Altered Siltstone do not contact each other.

**Figure 14.6 Contact Plot for Gold in Lower Argillite versus Tuff Domain**




## 14.3 VARIOGRAPHY

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

The model parameters are shown in Table 14.8 and Table 14.9 and the models for gold are included as Appendix C.

**Table 14.8 Summary of Semivariogram Parameters for Gold**

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

**Table 14.9 Summary of Semivariogram Parameters for Silver**

| Azimuth (°) | Dip (°) | $C_o$ | $C_1$ | $C_2$ | Short Range (m) | Long Range (m) |
|---|---|---|---|---|---|---|
| **Silver in Upper Argillite** | | | | | | |
| 130 | 0 | 0.16 | 0.10 | 0.26 | 12 | 90 |
| 040 | -43 | 0.16 | 0.10 | 0.26 | 12 | 50 |
| 220 | -47 | 0.16 | 0.10 | 0.26 | 22 | 40 |
| **Silver in Tuff** | | | | | | |
| 063 | 0 | 0.10 | 0.10 | 0.21 | 15 | 36 |
| 333 | 0 | 0.10 | 0.10 | 0.21 | 10 | 20 |
| 0o | -90 | 0.10 | 0.10 | 0.21 | 15 | 100 |

*table continues…*

| Azimuth (°) | Dip (°) | $C_o$ | $C_1$ | $C_2$ | Short Range (m) | Long Range (m) |
|---|---|---|---|---|---|---|
| **Silver in Lower Argillites** | | | | | | |
| 130 | 0 | 0.10 | 0.10 | 0.20 | 20 | 80 |
| 040 | -15 | 0.10 | 0.10 | 0.20 | 15 | 40 |
| 220 | -75 | 0.10 | 0.10 | 0.20 | 15 | 120 |
| **Silver in Altered Siltstone** | | | | | | |
| 140 | 0 | 0.14 | 0.08 | 0.12 | 20 | 120 |
| 050 | 0 | 0.14 | 0.08 | 0.12 | 15 | 60 |
| 000 | -90 | 0.14 | 0.08 | 0.12 | 15 | 60 |
| **Silver in North Zone Argillite** | | | | | | |
| 133 | 0 | 0.15 | 0.10 | 0.11 | 20 | 90 |
| 223 | -65 | 0.15 | 0.10 | 0.11 | 30 | 100 |
| 43 | -25 | 0.15 | 0.10 | 0.11 | 30 | 80 |
| **Silver in Waste** | | | | | | |
| Omni Directional | | 0.10 | 0.20 | 0.20 | 30 | 80 |

Semivariogram models were also developed for calcium, arsenic, and sulphur based on combining all domains. Nested spherical models were fit to both variables. The parameters for calcium, sulphur, and arsenic are shown in Table 14.10.

**Table 14.10 Summary of Semivariogram Parameters for Calcium, Sulphur and Arsenic**

| Azimuth (°) | Dip (°) | $C_o$ | $C_1$ | $C_2$ | Short Range (m) | Long Range (m) |
|---|---|---|---|---|---|---|
| **Calcium in all Domains** | | | | | | |
| 135 | 0 | 0.048 | 0.040 | 0.050 | 15 | 320 |
| 045 | -45 | 0.048 | 0.040 | 0.050 | 12 | 250 |
| 225 | -45 | 0.048 | 0.040 | 0.050 | 15 | 70 |
| **Sulphur in all Domains** | | | | | | |
| 135 | 0 | 0.20 | 0.20 | 0.24 | 15 | 300 |
| 045 | -45 | 0.20 | 0.20 | 0.24 | 30 | 150 |
| 225 | -45 | 0.20 | 0.20 | 0.24 | 15 | 40 |
| **Arsenic in all Domains** | | | | | | |
| 135 | 0 | 0.10 | 0.18 | 0.12 | 10 | 90 |
| 045 | -45 | 0.10 | 0.18 | 0.12 | 40 | 110 |
| 225 | -45 | 0.10 | 0.18 | 0.12 | 20 | 80 |

## 14.4 Block Model

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

- Lower Left Corner
  - 603125 E    Column size = 15 m    150 columns
  - 5826305 N    Row size = 15 m    217 rows
- Top of Model
  - 1450    Level size = 5 m    241 levels
- No Rotation.

## 14.5 Bulk Density

From the pre-2012 drill core, 2,155 measurements for specific gravity (SG) were taken using the weight in air – weight in water method. Samples were from drill core in holes 05-DDH-251 to 10-DDH-918 spread across the mineralized zone in all lithologies. Table 14.11 and Table 14.12 summarizes the results sorted first by lithology and then by gold grade. While there are slight differences in the various lithologies, there appears to be no correlation between SG and gold grade. As a result blocks within the block model were assigned a SG based on lithology. Blocks straddling two or more lithologies were assigned a weighted average SG.

**Table 14.11 Summary of Measured Specific Gravities Sorted by Lithology**

| Zone | No. of SGs | Minimum | Maximum | Average |
|---|---|---|---|---|
| Upper Argillite | 305 | 2.39 | 3.00 | 2.76 |
| Tuff | 382 | 2.46 | 3.02 | 2.79 |
| Siltstones | 443 | 2.42 | 3.30 | 2.78 |
| Lower Argillite | 625 | 2.50 | 3.11 | 2.76 |
| North Zone Argillite | 392 | 2.60 | 3.28 | 2.77 |
| Waste | 8 | 2.66 | 2.92 | 2.80 |
| **Total** | **2,155** | **2.39** | **3.30** | **2.77** |

**Table 14.12 Summary of Measured Specific Gravities Sorted by Gold Grade**

| Gold Grade Range | No. of SGs | Minimum | Maximum | Average |
|---|---|---|---|---|
| >0.0 < 0.10 | 1,456 | 2.42 | 3.30 | 2.77 |
| ≥ 0.10 > 0.25 | 308 | 2.56 | 3.28 | 2.77 |
| ≥ 0.25 > 0.50 | 149 | 2.63 | 2.96 | 2.77 |
| ≥ 0.50 > 0.75 | 58 | 2.60 | 3.11 | 2.80 |
| ≥ 0.75> 1.00 | 43 | 2.70 | 3.00 | 2.79 |
| ≥ 1.00> 5.00 | 133 | 2.39 | 3.11 | 2.78 |
| ≥ 5.00 | 8 | 2.70 | 2.90 | 2.78 |
| **Total** | **2,155** | **2.39** | **3.30** | **2.77** |

## 14.6 GRADE INTERPOLATION

Ordinary kriging (OK) was used to interpolate grades for gold, silver, calcium, sulphur, and arsenic into blocks with some proportion within the mineralized solids. In all cases the kriging exercise was completed in a series of four passes with the search ellipse for each pass being a function of the semivariogram ranges.

Grades for gold and silver were estimated into blocks containing some percentage of Upper Argillites using only composites from Upper Argillites. A similar hard boundary strategy was used for blocks containing some percentage of Siltstones and North Zone Argillites. For blocks containing some percentage of Tuffs or Lower Argillites the search ellipse was allowed to see samples from either domain (a soft boundary). Within Upper Argillites there was a pronounced change in bedding dip which was modelled by an inflection plane (see Figure 14.2). For blocks on the north side of this plane the search ellipse was steepened to find the required composites.

In all cases the first pass at resource estimation used a search ellipse with dimensions equal to one quarter of the semivariogram range in the three principal directions. A minimum of four composites were required to estimate the block. For blocks not estimated in the first pass, a second pass using one half the semivariogram ranges was completed. A third pass using the full semivariogram range and a fourth using twice the range completed the exercise. In all cases the maximum number of composites used was restricted to 12 with a maximum of 3 from any single drillhole allowed. In cases where a block containing two domains was estimated for one but not the other, a fifth pass was run to produce a grade for the other domain. A similar procedure was used for blocks estimated for gold but not for silver since there were fewer silver composites.

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

The kriging parameters for gold are tabulated in Table 14.13.

For calcium, sulphur and arsenic all data domains were combined since these variables were not as well sampled.

**Table 14.13 Kriging Parameters for Gold in all Domains**

| Domain | Pass | Number Estimated | Azimuth/ Dip (°) | Distance (m) | Azimuth/ Dip (°) | Distance (m) | Azimuth/ Dip (°) | Distance (m) |
|---|---|---|---|---|---|---|---|---|
| Upper Argillite South of Inflection Plane | 1 | 3,602 | 130/0 | 22.5 | 40/-43 | 20.5 | 220/-47 | 12.0 |
| | 2 | 13,845 | 130/0 | 45.0 | 40/-43 | 41.0 | 220/-47 | 24.0 |
| | 3 | 9,831 | 130/0 | 90.0 | 40/-43 | 82.0 | 220/-47 | 48.0 |
| | 4 | 4,920 | 130/0 | 180.0 | 40/-43 | 164.0 | 220/-47 | 96.0 |
| Upper Argillite North of Inflection Plane | 1 | 715 | 130/0 | 22.5 | 40/-70 | 20.5 | 220/-20 | 12.0 |
| | 2 | 4,938 | 130/0 | 45.0 | 40/-70 | 41.0 | 220/-20 | 24.0 |
| | 3 | 12,249 | 130/0 | 90.0 | 40/-70 | 82.0 | 220/-20 | 48.0 |
| | 4 | 24,941 | 130/0 | 180.0 | 40/-70 | 164.0 | 220/-20 | 96.0 |
| Tuff | 1 | 15,345 | 63/0 | 34.0 | 333/-58 | 25.0 | 153/-32 | 12.5 |
| | 2 | 42,498 | 63/0 | 68.0 | 333/-58 | 50.0 | 153/-32 | 25.0 |
| | 3 | 30,110 | 63/0 | 136.0 | 333/-58 | 100.0 | 153/-32 | 50.0 |
| | 4 | 31,279 | 63/0 | 272.0 | 333/-58 | 200.0 | 153/-32 | 100.0 |
| Siltstones | 1 | 620 | 140/0 | 16.0 | 50/0 | 10.0 | 0/-90 | 25.0 |
| | 2 | 5,289 | 140/0 | 32.0 | 50/0 | 20.0 | 0/-90 | 50.0 |
| | 3 | 28,746 | 140/0 | 64.0 | 50/0 | 40.0 | 0/-90 | 100.0 |
| | 4 | 35,175 | 140/0 | 128.0 | 50/0 | 80.0 | 0/-90 | 200.0 |
| Lower Argillite | 1 | 737 | 130/0 | 17.5 | 40/-15 | 3.8 | 220/-75 | 27.5 |
| | 2 | 10,251 | 130/0 | 35.0 | 40/-15 | 7.5 | 220/-75 | 55.0 |
| | 3 | 74,677 | 130/0 | 70.0 | 40/-15 | 15.0 | 220/-75 | 110.0 |
| | 4 | 190,797 | 130/0 | 140.0 | 40/-15 | 30.0 | 220/-75 | 220.0 |
| North Zone Argillites | 1 | 1,235 | 133/0 | 20.5 | 43/-25 | 10.0 | 223/-65 | 18.0 |
| | 2 | 12,946 | 133/0 | 41.0 | 43/-25 | 20.0 | 223/-65 | 36.0 |
| | 3 | 39,251 | 133/0 | 82.0 | 43/-25 | 40.0 | 223/-65 | 72.0 |
| | 4 | 98,601 | 133/0 | 164.0 | 43/-25 | 80.0 | 223/-65 | 144.0 |

## 14.7 CLASSIFICATION

Based on the study herein reported, delineated mineralization of the Property is classified as a resource according to the following definition from NI 43-101:

*In this Instrument, the terms "mineral resource", "inferred mineral resource", "indicated mineral resource" and "measured mineral resource" have the meanings ascribed to those terms by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines adopted by CIM Council on August 20, 2000, as those definitions may be amended from time to time by the Canadian Institute of Mining, Metallurgy, and Petroleum.*

*A **Mineral Resource** is a concentration or occurrence of natural, solid, inorganic or fossilized organic material in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.*

The terms Measured, Indicated and Inferred are defined in NI 43-101 as follows:

*A '**Measured Mineral Resource**' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.*

*An '**Indicated Mineral Resource**' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.*

*An '**Inferred Mineral Resource**' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information* *and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.*

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

For this resource estimate, in general blocks estimated during the first pass using a search ellipse with dimensions equal to one quarter of the semivariogram range were classified as measured. After this initial classification, the model was assessed and isolated blocks classified as measured were reclassified as indicated. Blocks estimated during the second pass, using one half the semivariogram ranges, were classified as Indicated. All other blocks were classified as Inferred.

The results are tabulated in Table 14.14 to Table 14.17 for the various classifications. A gold cut-off of 0.20 g/t has been highlighted based on the 2010 PEA (AGP 2010) as a possible open pit cut-off.

**Table 14.14 Spanish Mountain Measured Resource**

| Au Cut-off (g/t) | Tonnes > Cut-off (t) | Grade > Cut-off | | Contained Metal | |
|---|---|---|---|---|---|
| | | Au (g/t) | Ag (g/t) | Au (oz) | Ag (oz) |
| 0.10 | 40,310,000 | 0.48 | 0.65 | 620,000 | 840,000 |
| 0.15 | 34,600,000 | 0.53 | 0.66 | 590,000 | 730,000 |
| **0.20** | **29,360,000** | **0.60** | **0.67** | **560,000** | **630,000** |
| 0.25 | 24,880,000 | 0.66 | 0.65 | 530,000 | 520,000 |
| 0.30 | 21,240,000 | 0.73 | 0.64 | 500,000 | 440,000 |
| 0.40 | 15,940,000 | 0.86 | 0.64 | 440,000 | 330,000 |
| 0.50 | 12,060,000 | 0.99 | 0.65 | 380,000 | 250,000 |
| 0.60 | 9,290,000 | 1.12 | 0.65 | 340,000 | 190,000 |
| 0.70 | 7,300,000 | 1.25 | 0.66 | 290,000 | 150,000 |
| 0.80 | 5,920,000 | 1.37 | 0.67 | 260,000 | 130,000 |
| 0.90 | 4,850,000 | 1.49 | 0.68 | 230,000 | 110,000 |
| 1.00 | 4,030,000 | 1.60 | 0.68 | 210,000 | 90,000 |

### Table 14.15 Spanish Mountain Indicated Resource

| Au Cut-off (g/t) | Tonnes > Cut-off (t) | Grade > Cut-off | | Contained Metal | |
|---|---|---|---|---|---|
| | | Au (g/t) | Ag (g/t) | Au (oz) | Ag (oz) |
| 0.10 | 311,990,000 | 0.32 | 0.65 | 3,200,000 | 6,520,000 |
| 0.15 | 239,800,000 | 0.38 | 0.67 | 2,910,000 | 5,170,000 |
| **0.20** | **186,870,000** | **0.44** | **0.69** | **2,620,000** | **4,150,000** |
| 0.25 | 144,890,000 | 0.50 | 0.69 | 2,320,000 | 3,210,000 |
| 0.30 | 113,230,000 | 0.56 | 0.69 | 2,040,000 | 2,510,000 |
| 0.40 | 71,220,000 | 0.69 | 0.69 | 1,570,000 | 1,580,000 |
| 0.50 | 46,930,000 | 0.81 | 0.70 | 1,230,000 | 1,060,000 |
| 0.60 | 32,080,000 | 0.94 | 0.71 | 960,000 | 730,000 |
| 0.70 | 22,670,000 | 1.06 | 0.72 | 770,000 | 520,000 |
| 0.80 | 16,440,000 | 1.17 | 0.73 | 620,000 | 390,000 |
| 0.90 | 12,030,000 | 1.29 | 0.74 | 500,000 | 290,000 |
| 1.00 | 8,860,000 | 1.42 | 0.75 | 400,000 | 210,000 |

### Table 14.16 Spanish Mountain Measured Plus Indicated Resource

| Au Cut-off (g/t) | Tonnes > Cut-off (t) | Grade > Cut-off | | Contained Metal | |
|---|---|---|---|---|---|
| | | Au (g/t) | Ag (g/t) | Au (oz) | Ag (oz) |
| 0.10 | 352,290,000 | 0.34 | 0.65 | 3,820,000 | 7,360,000 |
| 0.15 | 274,400,000 | 0.40 | 0.67 | 3,500,000 | 5,910,000 |
| **0.20** | **216,220,000** | **0.46** | **0.68** | **3,180,000** | **4,730,000** |
| 0.25 | 169,770,000 | 0.52 | 0.68 | 2,850,000 | 3,710,000 |
| 0.30 | 134,470,000 | 0.59 | 0.68 | 2,540,000 | 2,940,000 |
| 0.40 | 87,160,000 | 0.72 | 0.68 | 2,010,000 | 1,910,000 |
| 0.50 | 58,990,000 | 0.85 | 0.69 | 1,610,000 | 1,310,000 |
| 0.60 | 41,370,000 | 0.98 | 0.70 | 1,300,000 | 930,000 |
| 0.70 | 29,970,000 | 1.10 | 0.71 | 1,060,000 | 680,000 |
| 0.80 | 22,360,000 | 1.23 | 0.72 | 880,000 | 520,000 |
| 0.90 | 16,870,000 | 1.35 | 0.72 | 730,000 | 390,000 |
| 1.00 | 12,900,000 | 1.47 | 0.73 | 610,000 | 300,000 |

Note: Tonnages and contained metals may not exactly equal individual tables due to rounding.

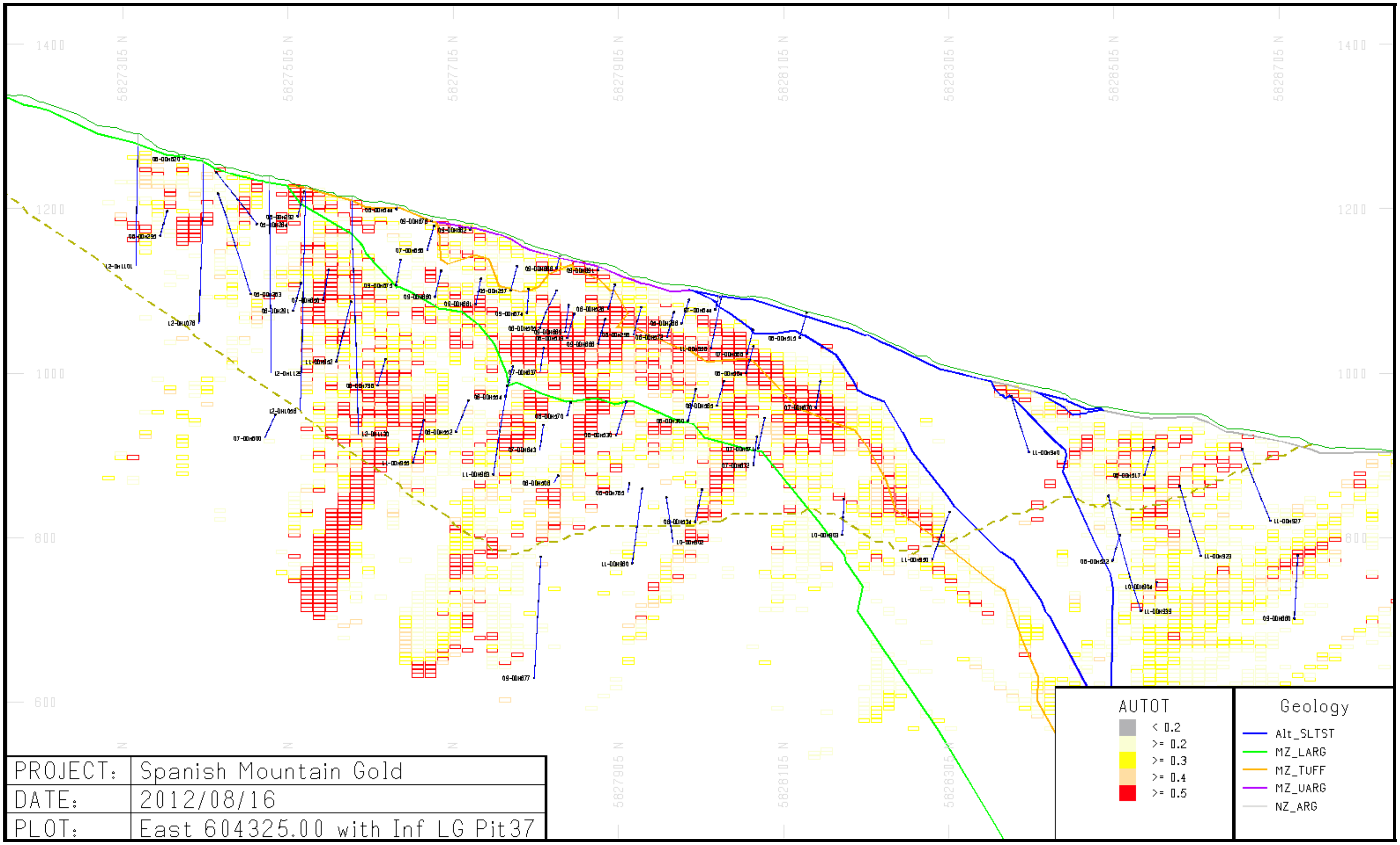
**Table 14.17 Spanish Mountain Inferred Resource**

| Au Cut-off (g/t) | Tonnes > Cut-off (t) | Grade > Cut-off | | Grade > Cut-off | |
|---|---|---|---|---|---|
| | | Au (g/t) | Au (g/t) | Au (oz) | Ag (oz) |
| 0.10 | 697,310,000 | 0.24 | 0.60 | 5,380,000 | 13,450,000 |
| 0.15 | 459,790,000 | 0.30 | 0.63 | 4,430,000 | 9,310,000 |
| **0.20** | **316,740,000** | **0.36** | **0.65** | **3,650,000** | **6,620,000** |
| 0.25 | 214,940,000 | 0.42 | 0.66 | 2,910,000 | 4,560,000 |
| 0.30 | 147,830,000 | 0.49 | 0.67 | 2,320,000 | 3,180,000 |
| 0.40 | 70,160,000 | 0.65 | 0.70 | 1,470,000 | 1,580,000 |
| 0.50 | 39,320,000 | 0.81 | 0.68 | 1,030,000 | 860,000 |
| 0.60 | 23,850,000 | 0.99 | 0.67 | 760,000 | 510,000 |
| 0.70 | 15,990,000 | 1.15 | 0.67 | 590,000 | 340,000 |
| 0.80 | 11,650,000 | 1.30 | 0.67 | 490,000 | 250,000 |
| 0.90 | 8,620,000 | 1.47 | 0.66 | 410,000 | 180,000 |
| 1.00 | 6,820,000 | 1.60 | 0.63 | 350,000 | 140,000 |

## 14.8 Model Verification

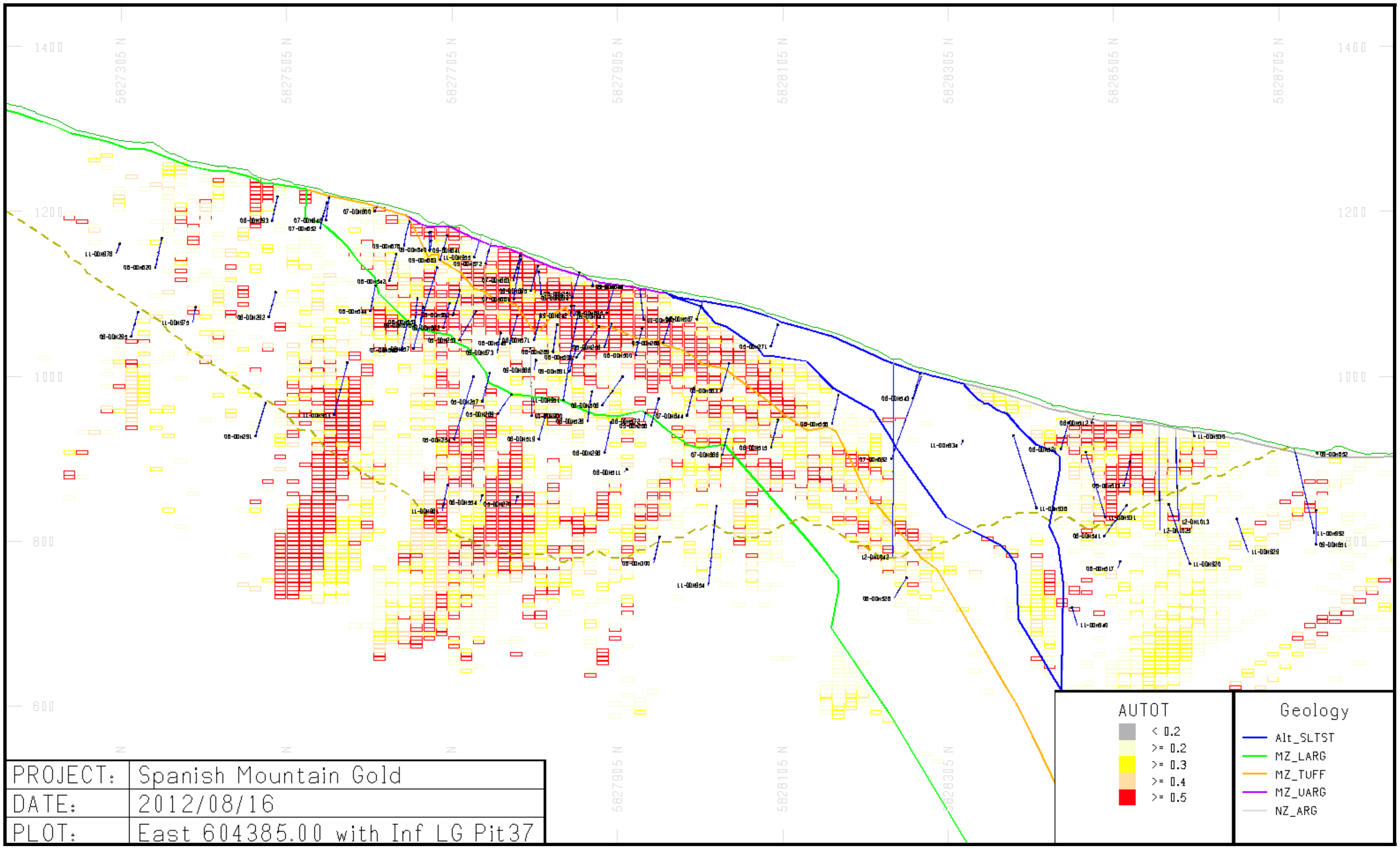
Detailed north-south cross sections were produced on 30 m intervals through the deposit showing gold grades with drillholes and geologic domain boundaries on one set and classification with drillholes and geologic domains on the other set. These cross sections were evaluated by SMG geologic staff to verify the model. The model was thought to be a valid estimation of grades that honoured the domains and the drillhole assays. Example cross sections are shown in Figure 14.7 to Figure 14.10 for 604325 east and 604385 east showing both gold grades and classification.

**Figure 14.7 Cross Section 604325 East**




Note: Looking west showing estimated gold grades, drillholes and geologic domains

**Figure 14.8 Cross Section 604385 East**




Note: Looking west showing estimated gold grades, drillholes and geologic domains

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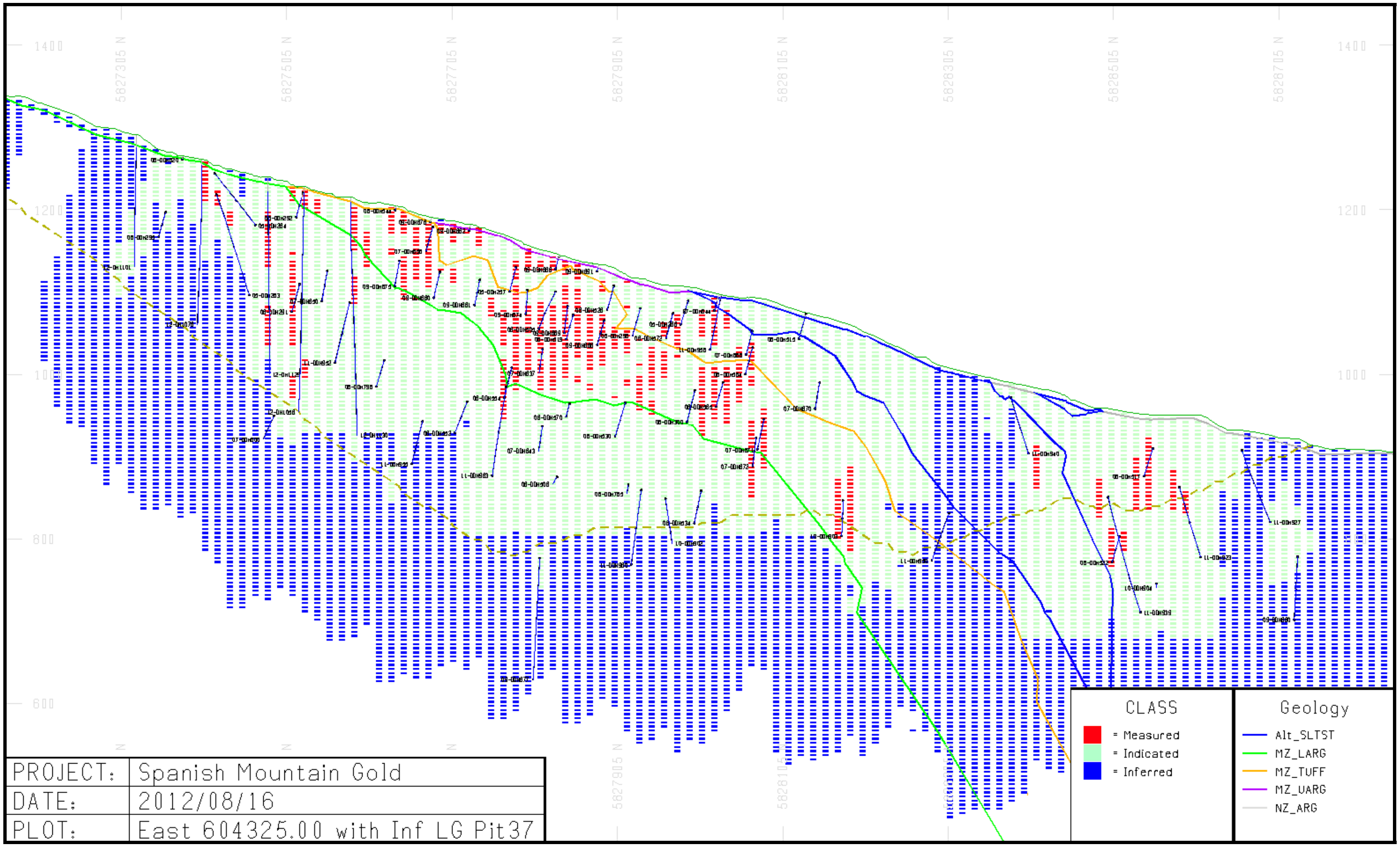
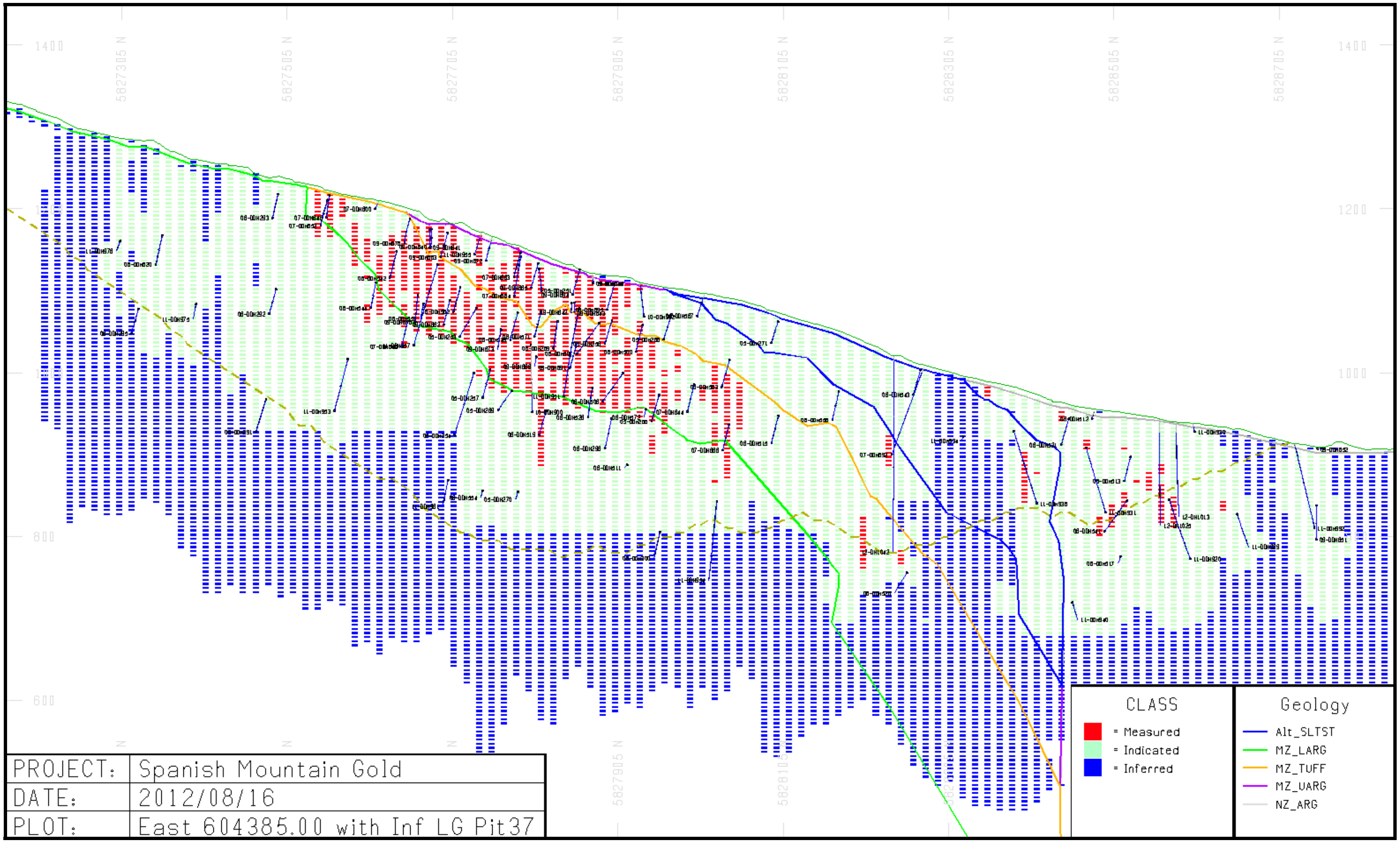
**Figure 14.9 Cross Section 604325 East**




Note: Looking west showing block classification

**Figure 14.10 Cross Section 604385 East**




Note: Looking west showing block classification

# 15.0 MINERAL RESERVE ESTIMATES

A mineral reserve has not been estimated for the Project as part of this PEA.

A mineral reserve is the economically mineable part of a Measured or Indicated Mineral Resource.

# 16.0 MINING METHODS

## 16.1 SUMMARY

The Spanish Mountain deposit will be mined using a conventional open pit mining method, using off-highway haul trucks and hydraulic shovels. The waste and mineralized material will be drilled and blasted, using typical grade control methods and blasthole sampling.

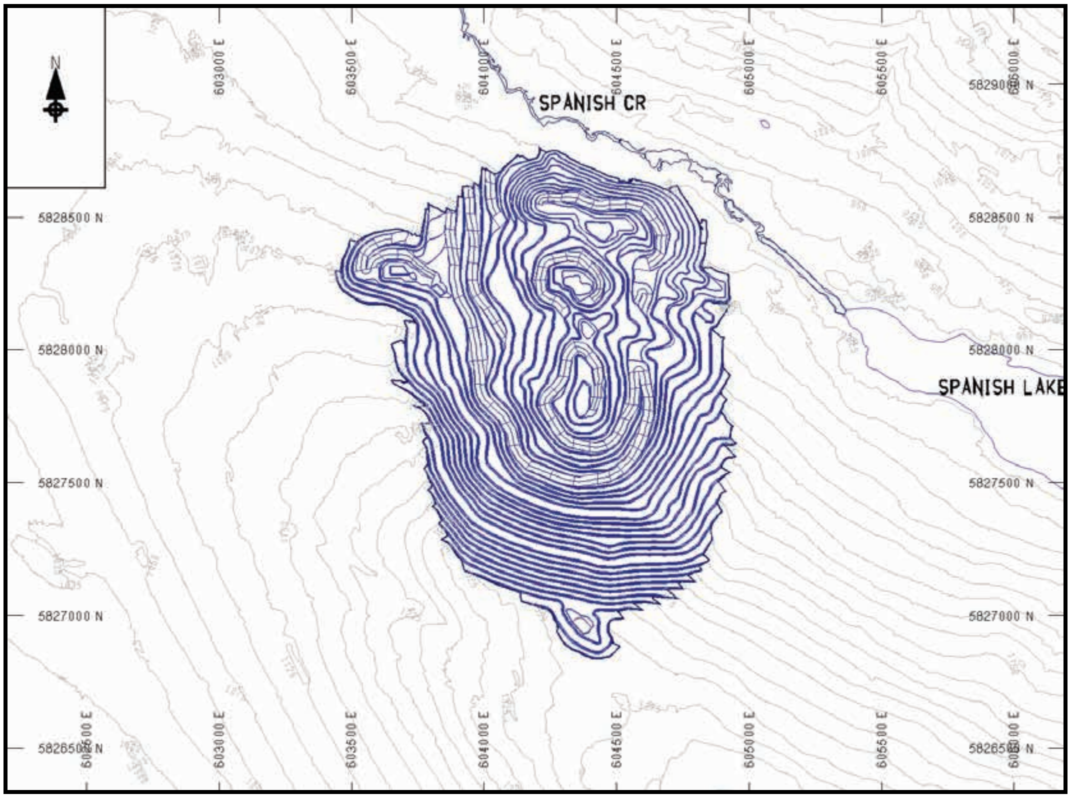
The open pit was designed for an approximately 14-year LOM. The potential in-pit resource, summarized in Table 16.1 on a 0.20 g/t gold cut-off, form the basis of the mine plan and production schedule. This estimate includes Inferred Resources that are considered too geologically speculative to warrant economic considerations that would enable them to be categorized as mineral reserves. There is no certainty that the economic results from this PEA will be realized based on this resource category.

**Table 16.1 Potential In-pit Resource Estimate**

| | Unit | Amount |
|---|---|---|
| Measured and Indicated Resource Class | kt | 167,198 |
| Gold Grade | g/t | 0.477 |
| Silver Grade | g/t | 0.675 |
| Inferred Resource Class | kt | 38,700 |
| Gold Grade | g/t | 0.498 |
| Silver Grade | g/t | 0.667 |
| Total All Classes | kt | 205,898 |
| Gold Grade | g/t | 0.481 |
| Silver Grade | g/t | 0.673 |
| Waste Material | kt | 464,874 |
| Strip Ratio | t/t | 2.3 |

The crusher will be fed with material from the pit and supplemented by the ROM stockpile in some years, at an 14-year LOM average rate of 40,000 t/d. During the last operating year (Year 15), remaining stockpiled material will be reclaimed and fed to the mill. Figure 16.1 shows a plan view of the preliminary design for the ultimate pit.

**Figure 16.1 Ultimate Pit Design – Plan View**



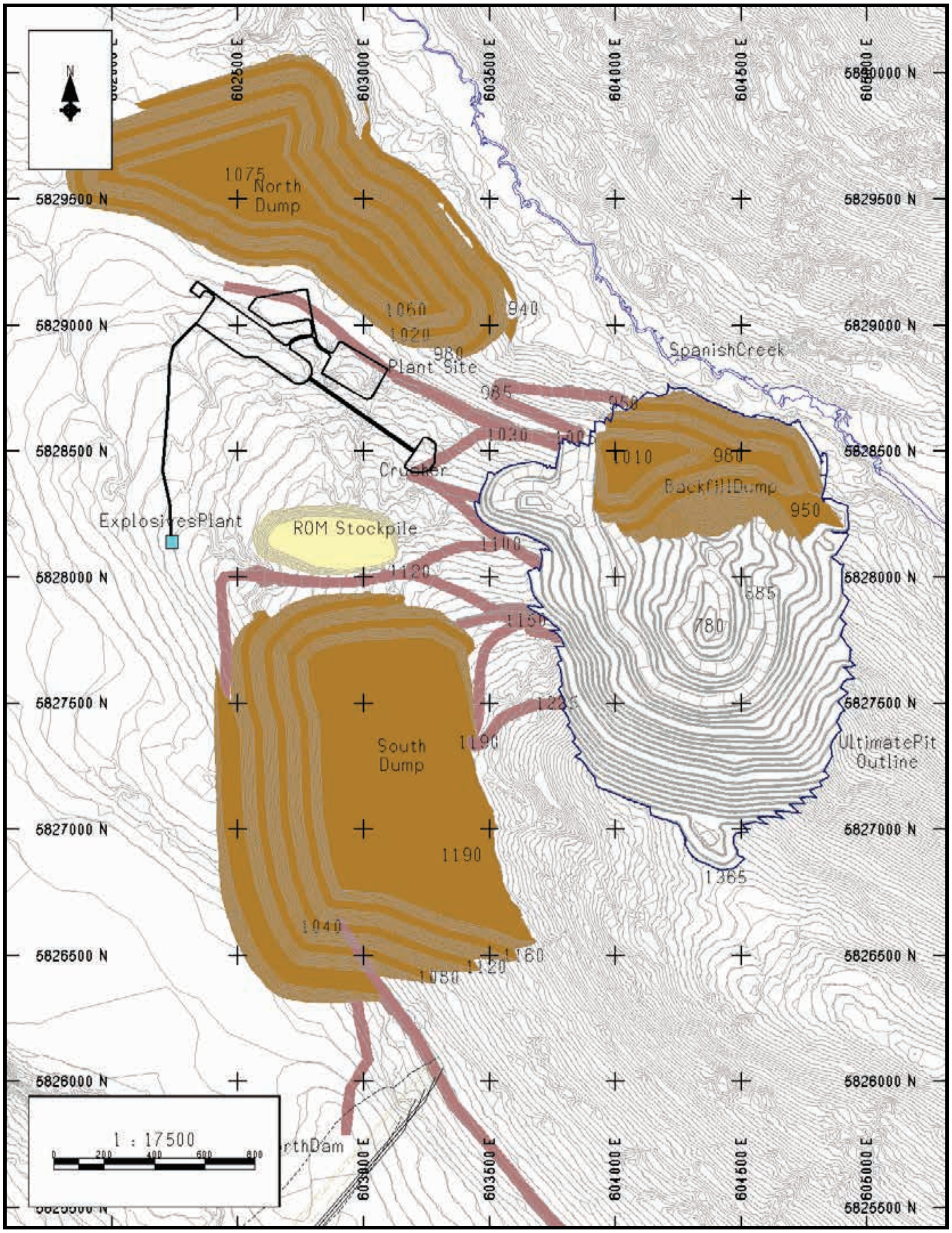

To develop the most economic feed to the mill in the early years, and to provide a smooth transitional stripping plan for the duration of the LOM, open pit mining is scheduled from six designed mining phases. Phase 1 will commence near the centre of the deposit, where the highest grade of mineralized resource and lowest strip ratio will be encountered.

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

The majority of the pit waste material will be hauled to waste dumps located on the west side of the pit. Some backfilling will also be available when the north end of the pit is completed later in the mine life. Preliminary geochemistry studies on the pit rock indicate that between 90 to 95% of the pit waste rock is non-potentially acid generating (NPAG). The remaining potentially acid generating (PAG) will be sub-aqueously disposed of in the tailings pond. A small amount of the PAG rock will be used for upstream construction of the dam embankment where it will be eventually submersed. Suitable NPAG pit waste rock will also be hauled to the tailings storage facility for dam construction, as needed.

Figure 16.2 shows the mine layout for the pit, waste dumps, and ROM stockpile.

**Figure 16.2 General Arrangement for Pit and Waste Dumps**



## 16.2 Open Pit Optimization Study

For the purposes of this study, MMTS applied a widely-accepted open pit optimization methodology for preliminary assessments. The potential resource for the open pit is initially evaluated by pit simulations on the geological resource model using the Lerchs-Grossman (LG) computer algorithm. Preliminary pit shells are generated from the simulations by varying input design parameters for sensitivity analyses. Selected shells with contained mineralized materials are assessed for suitability for determining the ultimate pit and framework development for the mining phases.

These simulated pit shells do not incorporate in-pit ramps, wall smoothing, and other details that result from the subsequent design phase of the study. Therefore, the results from the pit optimization study should not be relied upon except to provide an indication of potential mineral resources that must be validated by proper pit designs.

### 16.2.1 Design Parameters

Economic values were assigned on Measured, Indicated, and Inferred Resource classes as categorized in the resource block model. The input parameters applied on the base case optimized pit, as shown in Table 16.2, were estimated based on related studies and discussions with SMG. Costs, exchange rate, recoveries, and pit slope angles are preliminary and optimization-specific; they are not necessarily used for the financial analysis of the Project.

**Table 16.2 Base Case Design Parameters for Optimized Pit**

| Design Parameter | Unit | Amount |
|---|---|---|
| Exchange Rate | US$:CAD$ | 1.00 |
| Gold Price (LME) | US$/oz | 1,350 |
| Silver Price (LME) | US$/oz | 0 |
| Processing Cost | CAD$/t material feed | 5.78 |
| G&A Cost | CAD$/t material feed | 0.76 |
| Mining Cost at 950 m – Mineralized Material | CAD$/t material feed | 1.64 |
| Mining Cost at 950 m – Waste | CAD$/t waste | 1.76 |
| Incremental Bench Cost (below approximately 950 m) | CAD$/t | 0.015 |
| Stockpile and Re-handle Cost | CAD$/t | 0.60 |
| Process Recovery | % | Variable on Au head grade |
| Gold Refining Cost | US$/oz | 8.00 |
| Gold Refining Payable Factor | % | 99.5 |
| Silver Refining Payable Factor | % | 90 |

*table continues…*

| Design Parameter | Unit | Amount |
|---|---|---|
| Royalty Payment | % | 1.5 |
| Overall Pit Slope Angle with Ramp | degrees | Variable, BGC recommendations |
| Offset from Spanish Creek – Riparian Zone | metres | 915 contour south side of Spanish Creek |

The market gold price used for the base case is US$1,350/oz, below the three-year trailing average of $1,436/oz. The process recovery percentage was provided by SMG, based on the process recovery design described in Section 17.0. The recovery formula is:

$$1.3834 \times \ln(Au_{gm/t}) + 91.1$$, where Au is the mill head grade (gm/t).

### 16.2.2 Spanish Creek Restriction

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

### 16.2.3 Pit Optimization Variable – Net Metal Value

The pits were optimized based on the net metal value calculated from the gold grade item in the 3D block model. The silver grade was excluded from the calculation for this study as it represents an insignificant economic component. The net metal value is calculated as follows:

*Net metal value ($/t) gold = Au grade (g/t) x 1/31.1035 (oz/g) x payable gold (%) x net gold price ($/oz) at mine gate x process recovery (%)*

The parameters used to calculate the net metal value for the base case are provided in Table 16.3. The net metal value calculated for the base case is carried in the model as the NSR item value.

**Table 16.3 NSR Parameters**

| Parameter | Units | Amount |
|---|---|---|
| Gold Price (LME) | US$/oz | 1,350 |
| Refining Charges | US$/oz | 8.00 |
| Royalty Payments | % | 1.5 |
| Gold Payable | % | 99.5 |
| Net Gold Price (Mine Gate) | US$/oz | 1,315 |

## Pit Optimization Results

Table 16.4 lists the undiluted mineralized material and waste quantities contained in the optimized pit shells generated against various inputs for gold price. The base case is LG Shell 39, where the input gold price is US$1,350/oz, and contains 205 Mt of mineralized material at 2.3 to 1 strip ratio. The contained gold is 3.2 Moz at the cut-off grade of 0.20 g/t gold.

The other LG shells were generated to test the sensitivity of the resource, assess possible pit phases and the opportunities for expansion. Figure 16.3 provides a graphical illustration of contained mineralized material in the LG shells.

**Table 16.4 Contained Mineralized Material (0.2 g/t Gold Cut-off) – Gold Price Sensitivity**

| Gold Price (At Market) (US$/oz) | Variance from Base Case (%) | Mineralized Material (kt) | Au Grade (g/t) | Contained Au Metal (koz) | Waste (kt) | Strip Ratio (t:t) |
|---|---|---|---|---|---|---|
| 200 | -85.2 | 27 | 1.87 | 1.6 | 35 | 1.3 |
| 250 | -81.5 | 320 | 1.51 | 15.5 | 638 | 2.0 |
| 300 | -77.8 | 643 | 1.45 | 30.0 | 1,361 | 2.1 |
| 350 | -74.1 | 1,033 | 1.29 | 42.8 | 1,807 | 1.7 |
| 400 | -70.4 | 1,806 | 1.16 | 67.6 | 3,408 | 1.9 |
| 450 | -66.7 | 9,812 | 0.74 | 234.4 | 8,759 | 0.9 |
| 500 | -63.0 | 19,527 | 0.70 | 437.5 | 17,240 | 0.9 |
| 550 | -59.3 | 24,111 | 0.69 | 533.2 | 23,243 | 1.0 |
| 600 | -55.6 | 41,353 | 0.63 | 843.4 | 40,895 | 1.0 |
| 650 | -51.9 | 48,514 | 0.62 | 974.1 | 51,123 | 1.1 |
| 700 | -48.1 | 54,965 | 0.61 | 1,075.9 | 57,160 | 1.0 |
| 750 | -44.4 | 60,656 | 0.60 | 1,164.1 | 64,440 | 1.1 |
| 800 | -40.7 | 66,570 | 0.59 | 1,257.8 | 74,169 | 1.1 |
| 850 | -37.0 | 79,181 | 0.57 | 1,445.2 | 95,665 | 1.2 |
| 900 | -33.3 | 86,743 | 0.56 | 1,556.4 | 109,688 | 1.3 |
| 950 | -29.6 | 92,447 | 0.55 | 1,637.0 | 120,824 | 1.3 |
| 1,000 | -25.9 | 97,658 | 0.54 | 1,704.1 | 129,153 | 1.3 |
| 1,050 | -22.2 | 133,578 | 0.52 | 2,237.8 | 238,777 | 1.8 |
| 1,100 | -18.5 | 152,498 | 0.51 | 2,496.2 | 287,048 | 1.9 |
| 1,150 | -14.8 | 162,540 | 0.51 | 2,644.1 | 323,529 | 2.0 |
| 1,200 | -11.1 | 169,024 | 0.50 | 2,732.2 | 344,925 | 2.0 |
| 1,250 | -7.4 | 173,753 | 0.50 | 2,794.8 | 360,404 | 2.1 |
| 1,300 | -3.7 | 181,141 | 0.50 | 2,890.3 | 386,313 | 2.1 |
| **1,350** | **Base Case** | **205,205** | **0.49** | **3,210.1** | **481,468** | **2.3** |
| 1,400 | 3.7 | 209,229 | 0.49 | 3,265.1 | 499,620 | 2.4 |
| 1,450 | 7.4 | 229,371 | 0.48 | 3,515.1 | 581,442 | 2.5 |
| 1,500 | 11.1 | 237,442 | 0.47 | 3,618.6 | 620,451 | 2.6 |

*table continues…*

| Gold Price (At Market) (US$/oz) | Variance from Base Case (%) | Mineralized Material (kt) | Au Grade (g/t) | Contained Au Metal (koz) | Waste (kt) | Strip Ratio (t:t) |
|---|---|---|---|---|---|---|
| 1,600 | 18.5 | 250,426 | 0.47 | 3,780.1 | 682,850 | 2.7 |
| 1,700 | 25.9 | 268,167 | 0.46 | 4,000.5 | 778,832 | 2.9 |
| 1,800 | 33.3 | 277,051 | 0.46 | 4,101.9 | 827,168 | 3.0 |
| 2,000 | 48.1 | 293,269 | 0.46 | 4,293.9 | 935,334 | 3.2 |
| 2,500 | 85.2 | 329,556 | 0.44 | 4,709.6 | 1,269,643 | 3.9 |

**Figure 16.3 Sensitivity of Contained In-pit Mineralization to Gold Price**

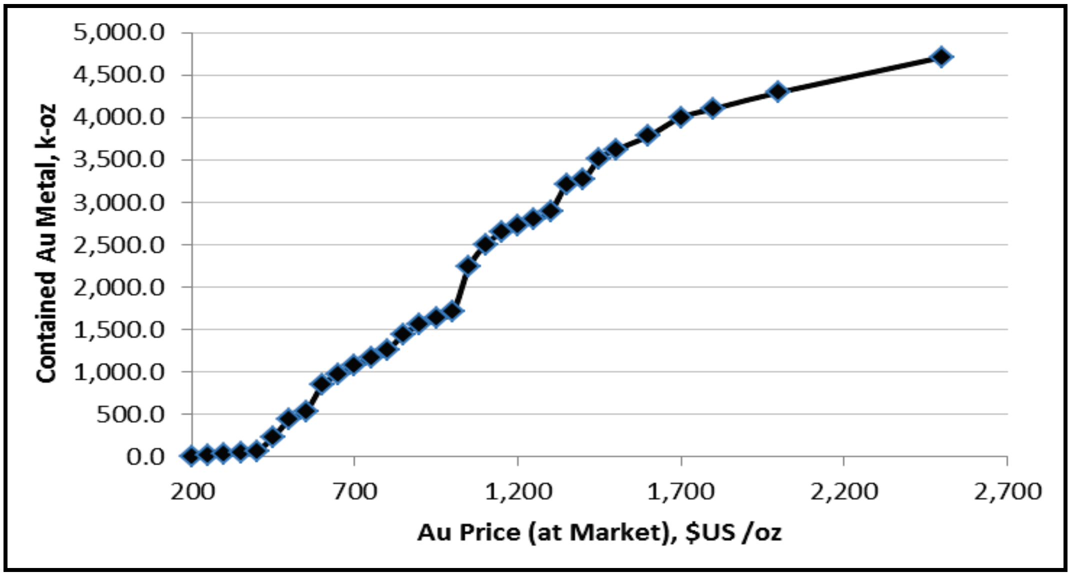



Table 16.5 and Figure 16.4 show the effect of changes to the average unit mining costs on mineralized material and waste quantities contained in the optimized pit shells. The assumed cost for the base case is $1.70/t for both mineralized material and waste. The sensitivity to an increase in unit mining cost to $1.80/t, where potentially contained gold resource is reduced by 22 Mt (or 303,000 oz gold), is significant. Therefore, it is important to maintain the average mine operating costs below $1.70/t.

**Table 16.5 Contained Mineralization (Cut-off on 0.20 g/t) – Mining Cost Sensitivity**

| Mining Costs $/t mined | Variance from Base Case (%) | Mineralized Material (kt) | Au Grade (g/t) | Contained Au Metal (koz) | Waste (kt) | Strip Ratio (t:t) |
|---|---|---|---|---|---|---|
| 1.50 | -11.8 | 225,264 | 0.48 | 3,466.7 | 566,990 | 2.5 |
| 1.60 | -5.9 | 208,710 | 0.49 | 3,259.4 | 498,409 | 2.4 |
| **1.70** | **Base Case** | **205,205** | **0.49** | **3,210.1** | **481,468** | **2.3** |
| 1.80 | 5.9 | 182,538 | 0.50 | 2,907.5 | 391,279 | 2.1 |
| 1.90 | 11.8 | 176,716 | 0.50 | 2,829.3 | 368,360 | 2.1 |
| 2.00 | 17.6 | 173,117 | 0.50 | 2,784.8 | 357,338 | 2.1 |
| 2.10 | 23.5 | 169,485 | 0.50 | 2,736.7 | 345,560 | 2.0 |
| 2.20 | 29.4 | 165,354 | 0.50 | 2,660.0 | 320,111 | 1.9 |
| 2.30 | 35.3 | 161,829 | 0.50 | 2,612.0 | 308,381 | 1.9 |

**Figure 16.4 Sensitivity of Contained In-pit Mineralization to Mining Costs**

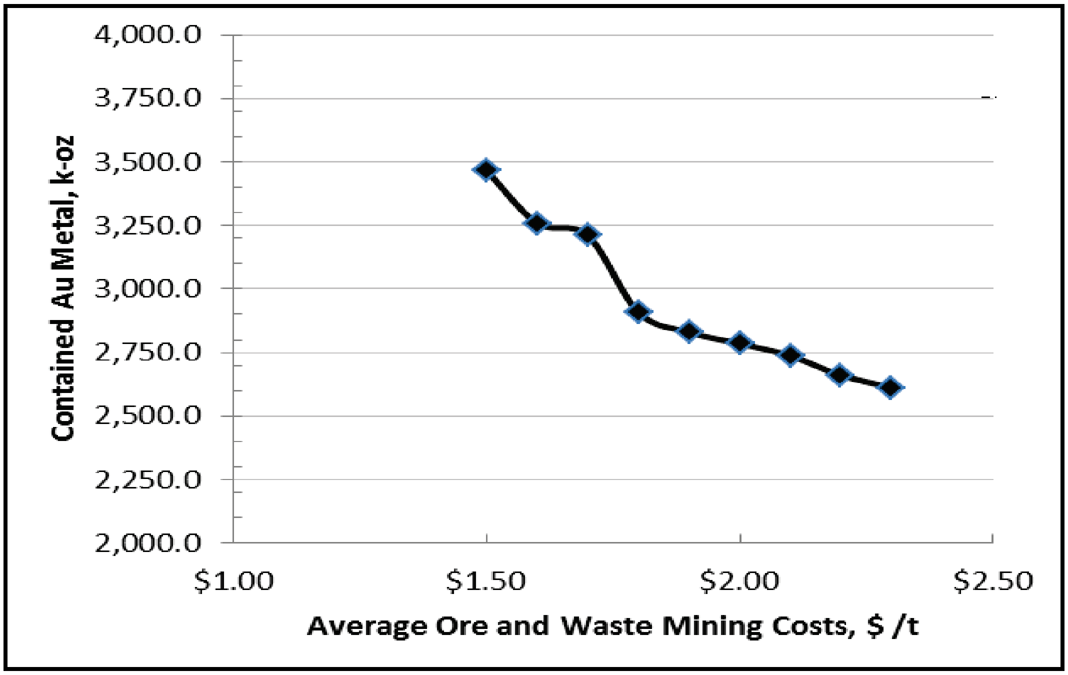



Figure 16.5 is a plan view showing the base case optimized pit. Figure 16.6 and Figure 16.7 are cross-sections through 604,385 east and 5,827,805 north, respectively, showing the block model resource within the base case pit shell ($1,350/oz) as well as the resource below it to the $2,500/oz simulated pit. The model blocks with hatching represent the Inferred Resource category.

**Figure 16.5 Base Case Optimized Pit – Plan View**

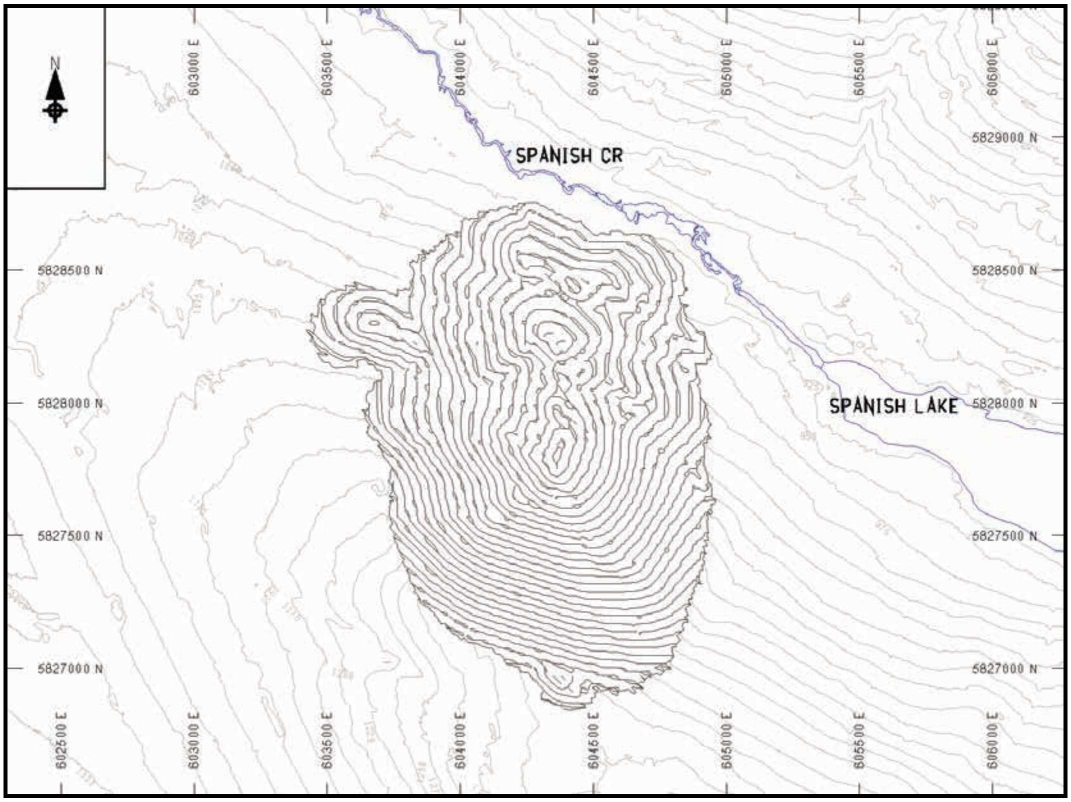



**Figure 16.6 Pit Simulations – Section 604,385 East (Looking West)**

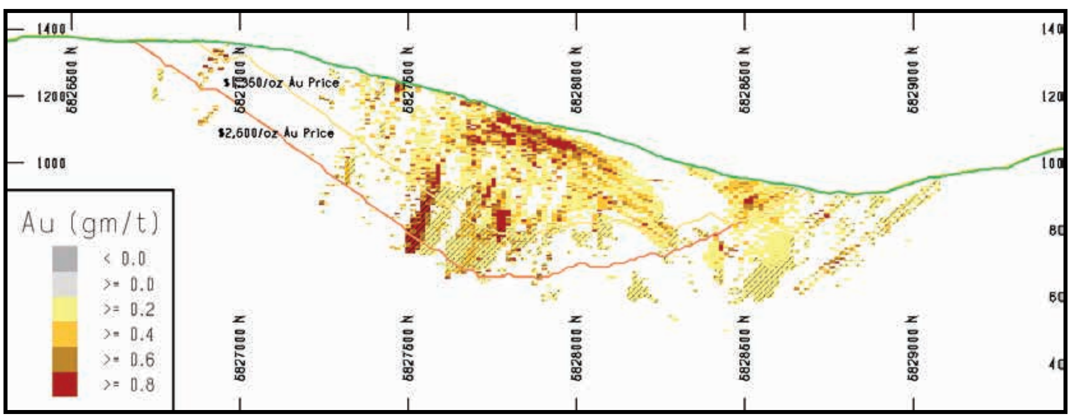
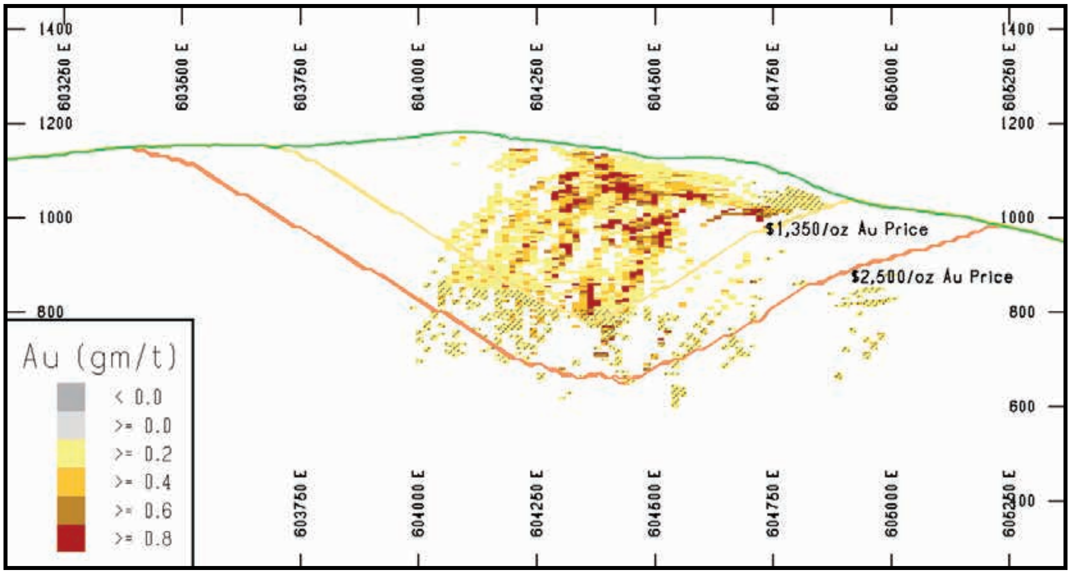


**Figure 16.7 Pit Simulations – Section 5,827,805 North (Looking North)**




## 16.3 Preliminary Pit Designs

The pit optimization indicated that the potential open pit resource approaches 205 Mt of mineralized material at a strip ratio of 2.3:1. Using this pit shell as a design guideline, an ultimate pit was created to incorporate high wall ramps, smooth pit walls, and workable mining phases. Figure 16.8 shows a plan view of the designed pit.

The designed pit dimensions are approximately 1.9 km long and 1.5 km wide at the widest section. The bottom of the pit is at 780 m elevation; the highest point is 1,360 m elevation, along the south end of the pit rim.

Figure 16.9 to Figure 16.12 are cross-sections showing the ultimate pit and the 3D block model gold grades. Figure 16.9 is a longitudinal north-south section approximately through the middle of the pit, while Figure 16.10 to Figure 16.12 are east-west sections through the pit. The model blocks with hatching represent the Inferred Resource category.

**Figure 16.8 Ultimate Pit Design – Plan View**

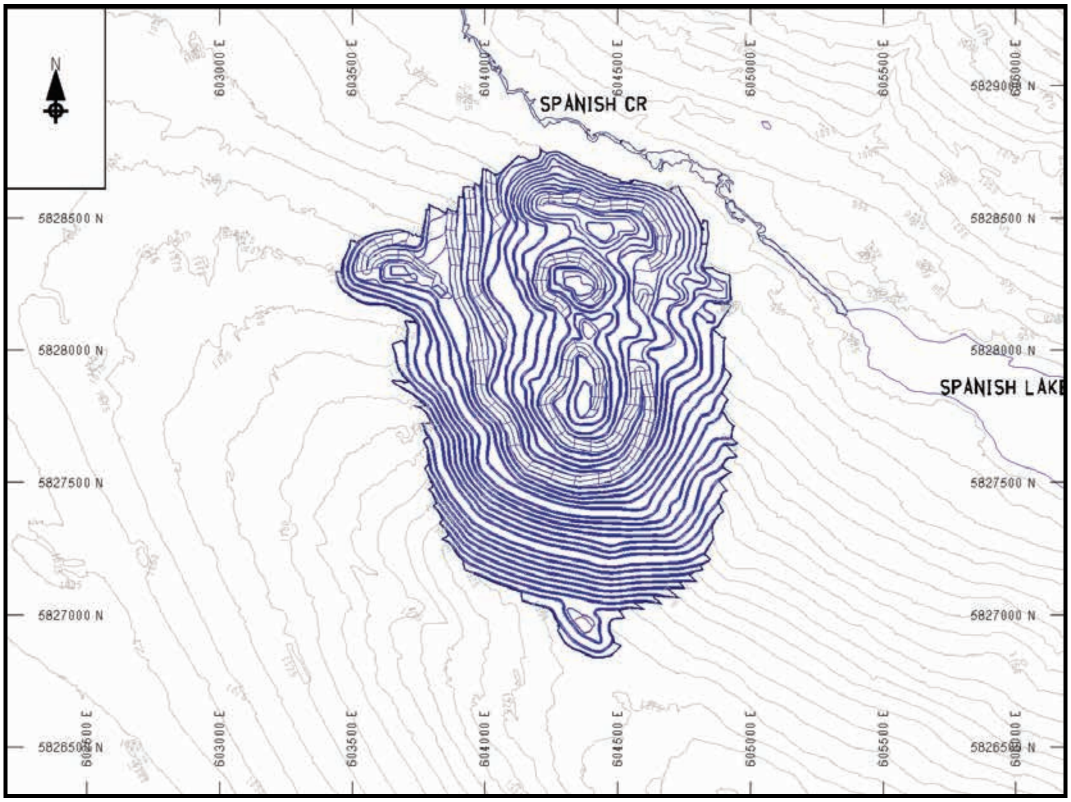



**Figure 16.9 Block Model Gold Values – Section 604,385 East (Looking West)**

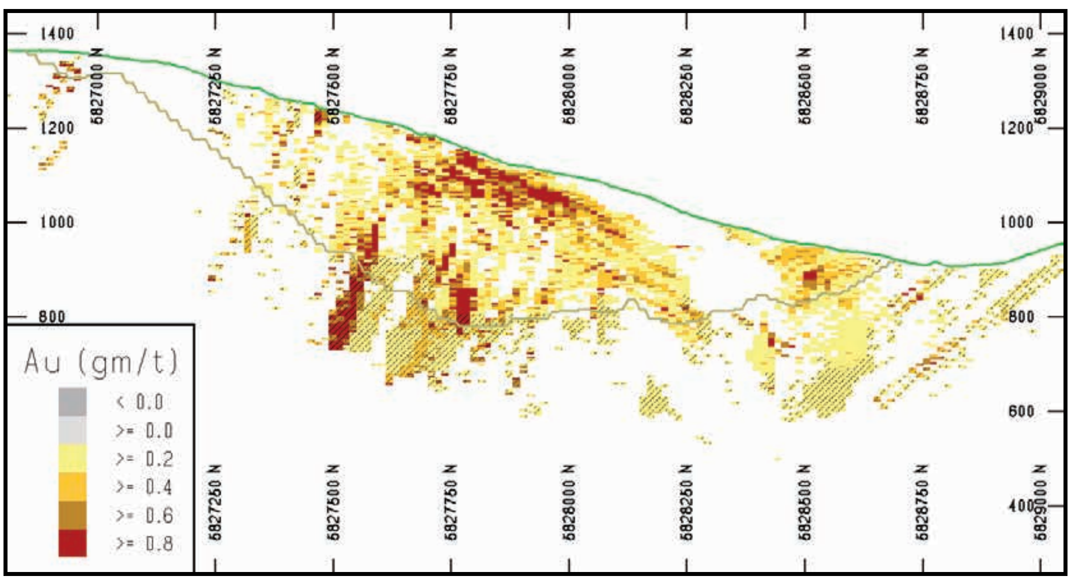


**Figure 16.10 Block Model Gold Values – Section 5,827,505 Looking North**

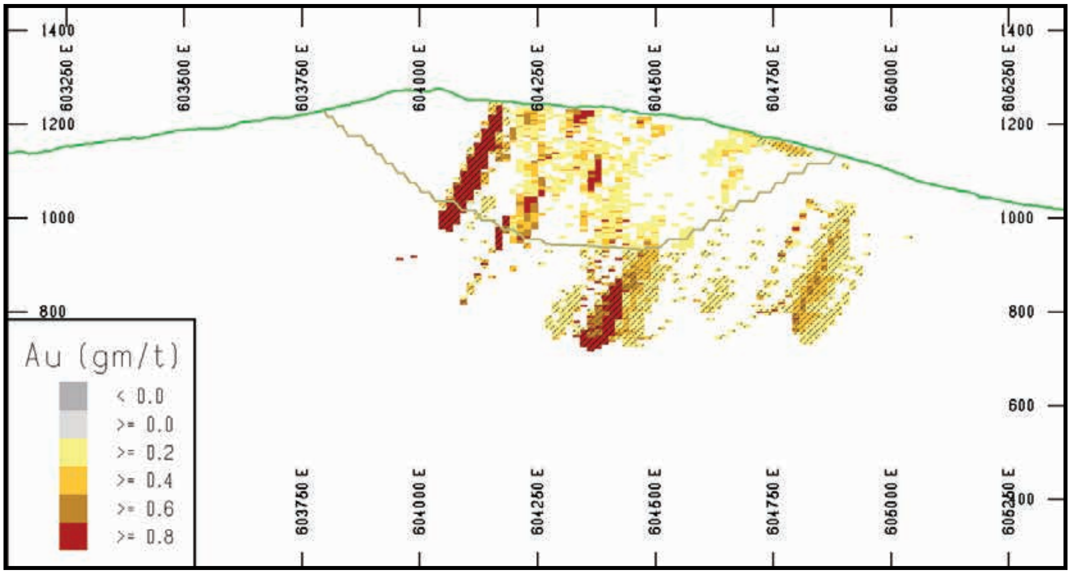



**Figure 16.11 Block Model Gold Values – Section 5,827,805 Looking North**

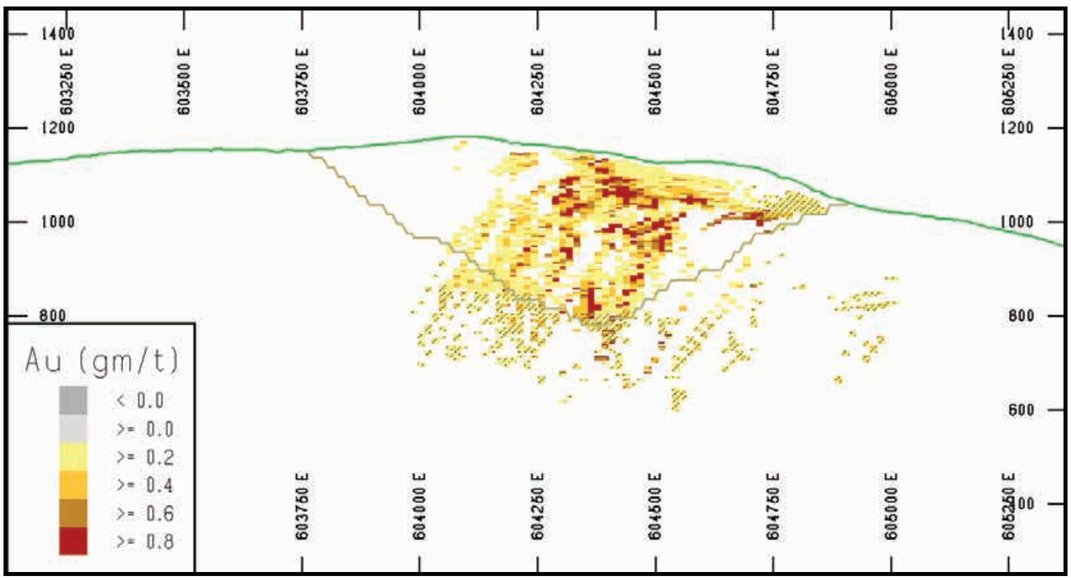
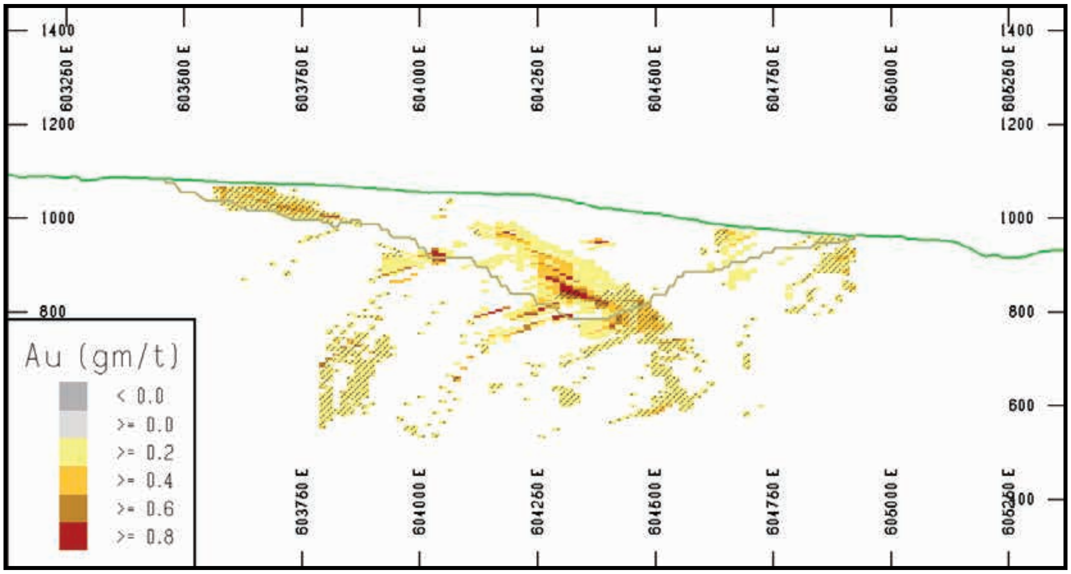


**Figure 16.12 Block Model Gold Values – Section 5,828,240 Looking North**




### 16.3.1 *Pit Slope Assumptions*

The pit slopes are designed based on preliminary recommendations developed from geotechnical drilling carried out in 2010 and 2011 (BGC 2012). The pit wall angles are limited generally by the orientations of the structural discontinuities in the rock mass and vary significantly depending on the design sector. Table 16.6 along with Figure 16.13 summarizes the wall design criteria for each sector. BGC's report in its entirety is provided in Appendix D. In-pit ramps (37 m wide) and geotechnical berms (minimum 20 m wide) are included in the design where necessary to reduce the interberm slope heights and facilitate geotechnical instrumentation and dewatering.

Groundwater pressures will have a significant effect the stability of the pit slopes. Preliminary hydrogeological studies carried out (BGC 2012) indicate that significant dewatering efforts will be required to depressurize the open pit slopes. To achieve the recommended pit slope angles a pit dewatering program consisting of vertical depressurization wells along the perimeter prior to and during excavation of the pit, augmented with horizontal drains in the pit walls during mining. Further studies will be necessary to finalize a pit dewatering plan and evaluate the impacts of the open pit on the regional water balance.

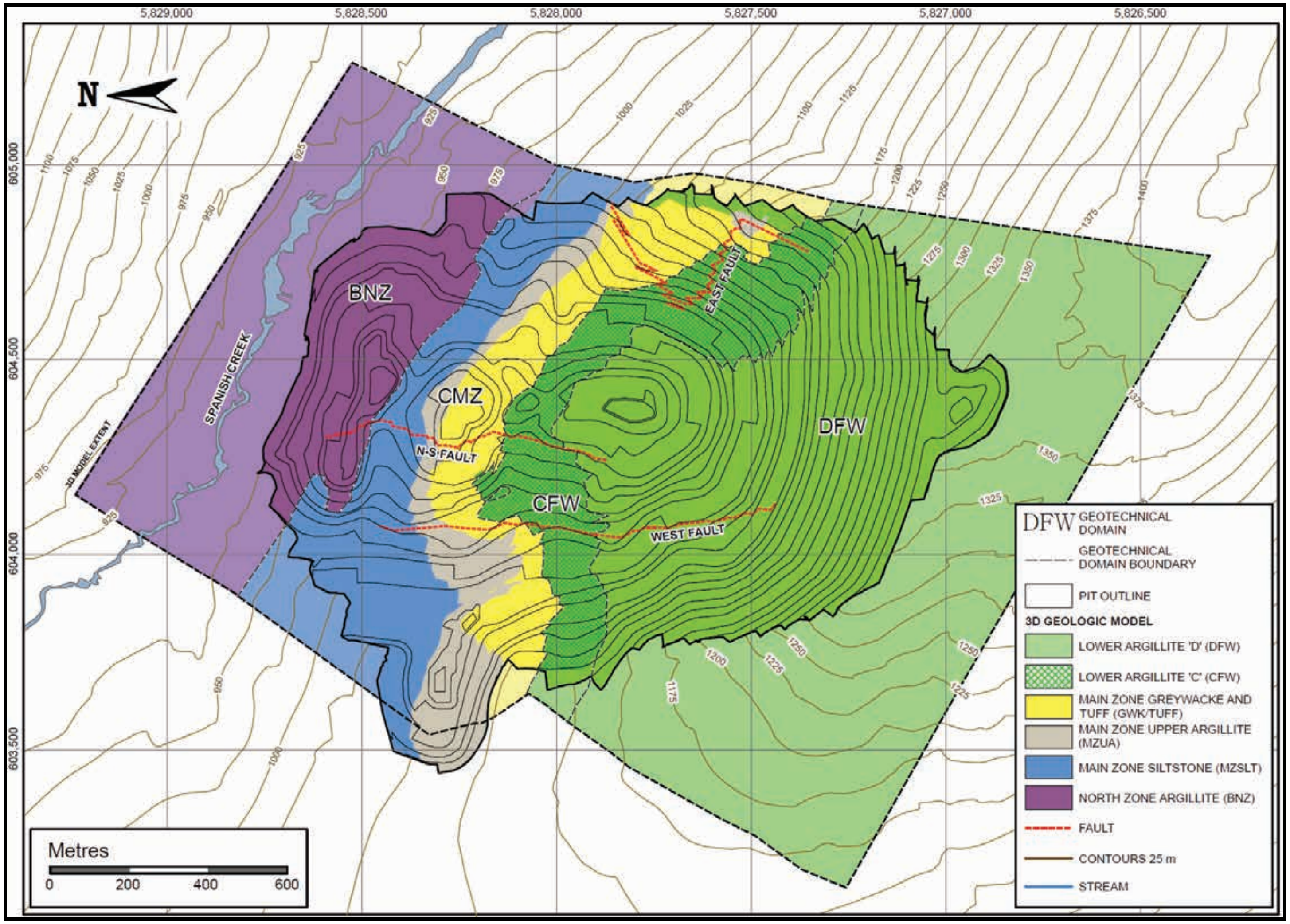
**Table 16.6 Pit Slope Design Recommendations**

| Domain | Design Sector | Slope Azimuth Start (°) | Slope Azimuth End (°) | Catch Bench Geometry: Design Height BH (m) | Catch Bench Geometry: Face Angle BA (°) | Catch Bench Geometry: Width BW (m) | Interberm Geometry: Maximum Height IBH (m) | Interberm Geometry: Angle IBA (°) | Design Control |
|---|---|---|---|---|---|---|---|---|---|
| BNZ | BNZ-180 | 155 | 205 | 10 | 65 | 18.6 | 100 | 23 | FB1-FC1 |
| BNZ | BNZ-220 | 205 | 235 | 10 | 65 | 15.7 | 100 | 26 | P-FC1 |
| BNZ | BNZ-300 | 235 | 005 | 10 | 65 | 9.0 | 100 | 36 | Bench geometry |
| BNZ | BNZ-020 | 005 | 035 | 10 | 65 | 10.7 | 100 | 33 | P-FA2 |
| BNZ | BNZ-046 | 035 | 057 | 10 | 65 | 15.7 | 100 | 26 | FB2-FA2 |
| BNZ | BNZ-089 | 057 | 120 | 10 | 65 | 17.6 | 100 | 24 | FB2-FA2 |
| BNZ | BNZ-130 | 120 | 140 | 10 | 65 | 14.1 | 100 | 28 | P-FB2, FB1-FC1 |
| BNZ | BNZ-148 | 140 | 155 | 10 | 65 | 15.7 | 100 | 26 | FB1-FC1 |
| CMZ | CMZ-2183 | 160 | 275 | 20 | 65 | 9.9 | 160 | 46 | T-FA3 |
| CMZ | CMZ-293 | 275 | 310 | 20 | 65 | 27.4 | 160 | 29 | P-BFD1 |
| CMZ | CMZ-339 | 310 | 007 | 20 | 65 | 34.2 | 160 | 25 | FA3-BFD1 |
| CMZ | CMZ-031 | 007 | 055 | 20 | 65 | 28.9 | 160 | 28 | P-BFA1 |
| CMZ | CMZ-090 | 055 | 125 | 20 | 65 | 34.2 | 160 | 25 | BFA1-FB2 |
| CMZ | CMZ-133 | 125 | 140 | 20 | 65 | 28.9 | 160 | 28 | BFA1-FB2 |
| CMZ | CMZ-150 | 140 | 160 | 20 | 65 | 20.8 | 160 | 34 | BFA1-FB2 |
| CFW | CFW-183 | 135 | 230 | 20 | 65 | 23.2 | - | 32 | Multiple wedges |
| CFW | CFW-268 | 230 | 305 | 20 | 65 | 36.2 | 160 | 24 | P-FC1 |
| CFW | CFW-320 | 305 | 335 | 20 | 65 | 25.9 | - | 30 | BFA1-FD1 |
| CFW | CFW-005 | 335 | 035 | 20 | 65 | 34.2 | - | 25 | BFA1-FD1 |
| CFW | CFW-055 | 035 | 075 | 20 | 65 | 28.9 | - | 28 | P-BFA1 |
| CFW | CFW-088 | 075 | 105 | 20 | 65 | 13.0 | 160 | 42 | Rockmass stability |
| CFW | CFW-120 | 105 | 135 | 20 | 65 | 16.6 | - | 38 | P-FB2 |
| DFW | DFW-188 | 155 | 220 | 20 | 65 | 13.9 | 160 | 41 | Multiple wedges |
| DFW | DFW-2353 | 220 | 250 | 20 | 65 | 9.5 | 160 | 47 | Bench geometry |
| DFW | DFW-261 | 250 | 272 | 20 | 65 | 12.2 | 160 | 43 | P-FD1 |
| DFW | DFW-284 | 272 | 295 | 20 | 65 | 19.1 | 160 | 35 | FD2-FC1 |
| DFW | DFW-320 | 295 | 345 | 20 | 65 | 23.2 | 160 | 32 | FD2-FC1 |
| DFW | DFW-025 | 345 | 065 | 20 | 65 | 36.2 | 160 | 24 | FA1-FD2 |
| DFW | DFW-103 | 065 | 140 | 20 | 65 | 28.9 | 160 | 28 | FA1-FB2 |
| DFW | DFW-148 | 140 | 155 | 20 | 65 | 20.8 | 160 | 34 | FA1-FB2 |

Notes: BH = Bench Height; BA = Bench Angle; BW = Bench Width; IBH = Intermediate Bench Height; IBA = Intermediate Bench Angle

Geotechnical berms (minimum 20 m) must be added to the slopes every 160 m in the Main Zone, and 100 m in the North Zone. Refer to Figure 16.13 for geotechnical domain boundaries. Potential for bench scale toppling will need to be addressed at future stages of design and may reduce achievable inter-ramp slope angles.

**Figure 16.13 Structural Domains for Pit Slope Design**



### 16.3.2 IN-PIT RESOURCE POTENTIAL

The potential resource estimate by classification contained within the designed ultimate pit above 0.20 g/t gold grade is summarized in Table 16.7.

**Table 16.7 Potential In-Pit Resource**

| | Unit | Amount |
|---|---|---|
| Measured and Indicated Resource Class | kt | 167,198 |
| Au Grade | g/t | 0.477 |
| Ag Grade | g/t | 0.675 |
| Inferred Resource Class | kt | 38,700 |
| Au Grade | g/t | 0.498 |
| Ag Grade | g/t | 0.667 |
| Total All Classes | kt | 205,898 |
| Au Grade | g/t | 0.481 |
| Ag Grade | g/t | 0.673 |
| Waste Material | kt | 464,874 |
| Strip Ratio | t/t | 2.3 |

Note: Table 16.7 includes Inferred Resources that are considered too geologically speculative to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that these data will be realized.

### 16.3.3 CUT-OFF GRADE

The cut-off grade is the estimated minimum grade of the mineralized material contained in the pit that is sufficient to cover the cost of milling, G&A, stockpile handling, stockpile maintenance, and the incremental haul costs from the waste dump to the crusher. For the purposes of this study, the cut-off grade was estimated to be 0.20 g/t (rounded up from 0.19 g/t) at the current gold price of $1,350/oz. Silver is currently considered an insignificant component and was not included in the calculation.

In general, the lower-grade mineralized material mined in the initial years will be stockpiled near the crusher, and then reclaimed and blended with the pit feed where possible. The remaining material will be reclaimed after the pit is completed, in Year 15.

### 16.3.4 DILUTION

The mining equipment selected for this study is sized to achieve relatively high mining rates and low unit mine operating costs; the fleet will also have some ability to effectively separate economic mineralized material from waste rock. The waste and mineralized material will require blasting; blasthole sampling methods will be implemented for mineralized material control. Some external dilution and losses will occur when waste mixes with mineralized material during normal blasting and excavation activities.

The mineralized material is represented in the resource model on a whole block basis. The 15 m x 15 m x 5 m size blocks are large mining units; averaging of the metal grade over an entire block suggests that the grade may be considerably smoothed resulting in significant internal model dilution.

For the purposes of this study, MMTS assumed that the selected mining fleet will effectively extract the mineralized material from the waste rock, and that mining dilution under normal situations will be offset by the modelling dilution. A value of 1% mining loss and dilution was applied to account for operating challenges and inefficiencies such as excessive blast heave, carry-back in truck boxes due to wet material, misdirected materials, and other unforeseen exceptions.

A thorough modelling evaluation and geostatistical analysis is necessary to better understand and quantify the internal dilution. This analysis will be undertaken at the next level of study.

## 16.4 PIT DEVELOPMENT PHASES

The mine plan incorporates pit phases, designed to advance mining activities that will generate maximum cash flow in the initial years, and balanced push-backs for smooth transitional waste stripping quantities thereafter.

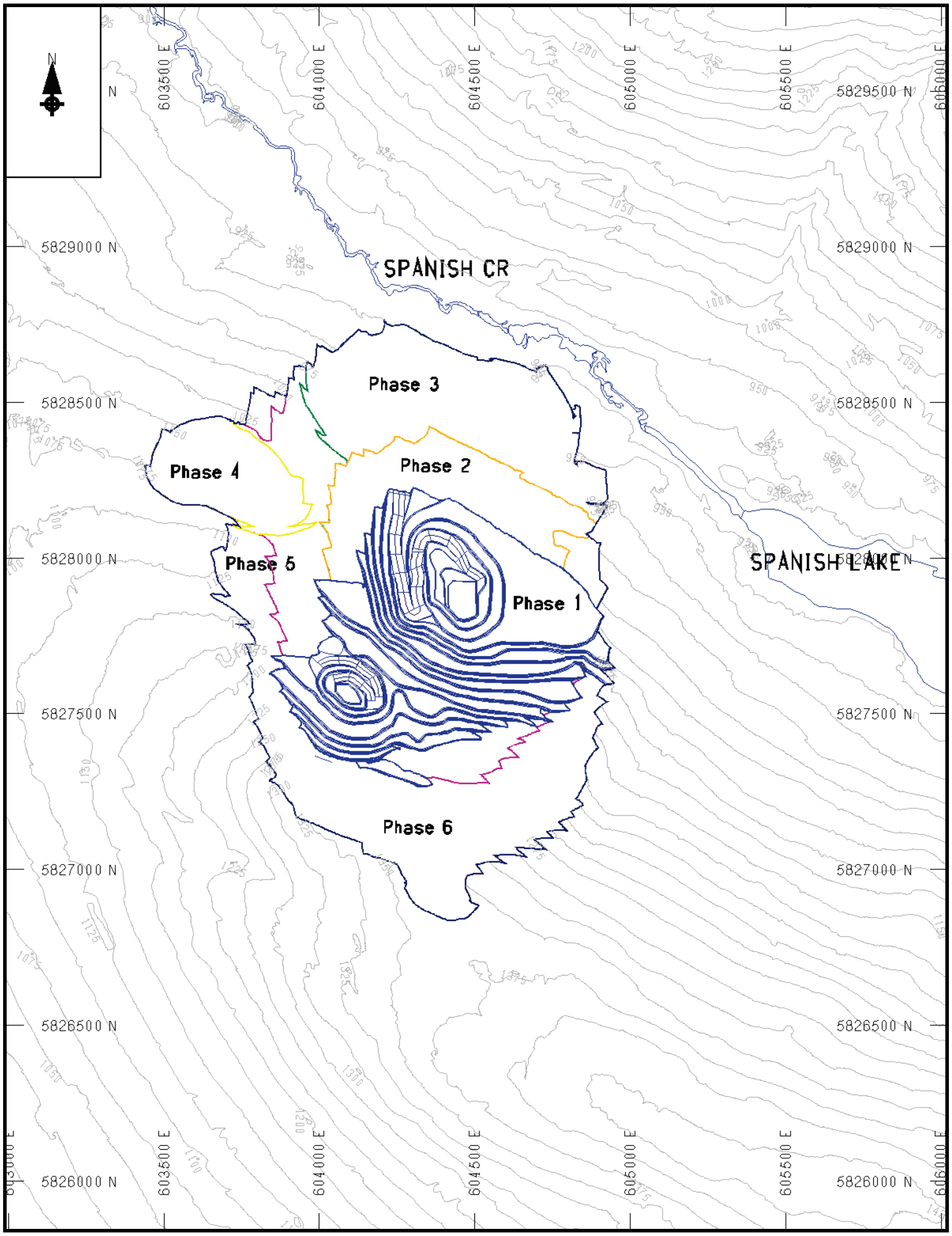
The starter pit, or the Phase 1 pit, will be located in the central zone of the ultimate pit. This zone contains the areas with minimum waste stripping and higher gold grades. Phase 2 will involve pushing back to the north and Phase 3 will involve mining to the north limit of the ultimate pit. Phase 4 will involve the construction of a small isolated pit on the west side, and Phase 5 will involve pushing back to the west, along with deepening of the pit. Phase 6 will involve mining the top of the ridge, and pushing the wall to its final limits on the west and south sides.

The potential pit resources by phase are shown in Table 16.8. Figure 16.14 is a plan plot showing the Phase 1 design along with outlines of Phases 2 through 6.

**Table 16.8 Potential Resource by Pit Phase**

| | Mineralized Material (kt) | Au Grade (g/t) | Ag Grade (g/t) | Waste (kt) | Strip Ratio (t/t) |
|---|---|---|---|---|---|
| Phase 1 | 42,513 | 0.600 | 0.683 | 44,173 | 1.0 |
| Phase 2 | 20,437 | 0.452 | 0.750 | 34,561 | 1.7 |
| Phase 3 | 22,879 | 0.394 | 0.792 | 37,961 | 1.7 |
| Phase 4 | 4,180 | 0.410 | 0.460 | 8,734 | 2.1 |
| Phase 5 | 45,886 | 0.444 | 0.662 | 114,507 | 2.5 |
| Phase 6 | 69,999 | 0.475 | 0.627 | 224,938 | 3.2 |
| **Total** | **205,894** | **0.481** | **0.673** | **464,874** | **2.3** |

**Figure 16.14 Pit Phase Development – Plan View**



## 16.5 WASTE ROCK MANAGEMENT FACILITY

### 16.5.1 WASTE ROCK CHARACTERIZATION

Sulphur, calcium, and arsenic values were interpolated into the 3D block model quantify the acid rock drainage (ARD) generating potential of the pit material. SRK Consulting (Canada) Inc. (SRK) provided the following formulas and criteria to categorize the pit waste rock.

*Acid Potential (AP) for block = 31.25 X S*

*Neutralization Potential (NP) for block = 37 X Ca + 8.8*

*where "S" is the sulphur value in percent and Ca is the calcium value in percent.*

The ARD generating categories are defined as follows:

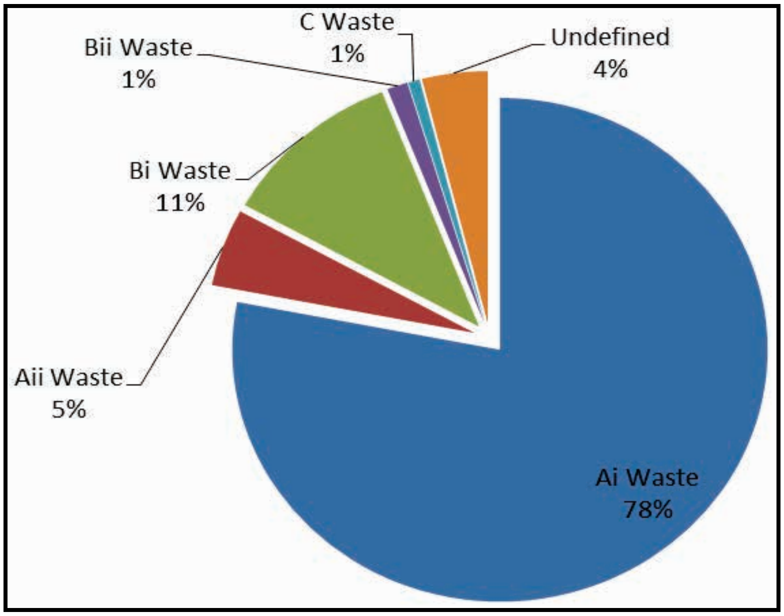
- Ai: if NP/AP ratio > 2 and arsenic < 150 ppm;
  unlikely to generate ARD and low potential for arsenic leaching – i.e. unlikely to require management.
- Aii: if NP/AP ratio > 2 and arsenic > 150;
  unlikely to generate ARD, arsenic leaching potentially significant.
- Bi: if NP/AP ratio > 1 and <= 2, and arsenic < 150;
  unlikely to generate ARD and low potential for arsenic leaching – i.e. unlikely to require management.
- Bii: if NP/AP ratio > 1 and <= 2 and arsenic > 150;
  same as Bi for ARD, but arsenic leaching potentially significant.
- C: if NP/AP ratio <= 1 for all arsenic values;
  PAG, very likely to require management.

These categories were written into an item in the 3D block model to characterize each block.

SRK's memo to SMG regarding the ML/ARD block model criteria is provided in Appendix E.

Figure 16.15 shows the distribution of the waste rock ARD categories contained in the ultimate pit. The majority of waste is Ai (78%) and Bi (11%), which are categorized as non-acid generating potential. The undefined category indicates that values for either, or all of, sulphur, calcium, or arsenic are not interpolated into the model blocks, likely due to missing data.

**Figure 16.15 Pit Waste Rock Distribution by ARD Characterization**




### 16.5.2 *WASTE ROCK SUITABILITY*

The possible waste rock destinations are determined by their ARD generating characterization. There are three destinations:

- the waste dumps on surface
- the tailings dam embankment
- the tailings pond for sub-aqueous disposal.

The destinations for each of the ARD categories are as follows:

- Ai: Non ARD, can be placed anywhere (e.g. waste dumps or tailings embankment).
- Aii: Potential ARD, requires management (e.g. sub-aqueous placement within no specific time frame, and can be used for upstream dam construction).
- Bi: Non ARD, can be placed anywhere (e.g. waste dumps or tailings embankment).
- Bii: Potential ARD, requires management – sub-aqueous placement after one year and assumed that it will be directed immediately to the tailings pond and co-mingled.
- C: ARD, required to be sub-aqueously placed immediately – will go to the tailings pond and co-mingled.

It is assumed that 75% of the undefined category of waste rock will have the same characteristic as Ai categorized material, while the remaining 25% will be similar to Aii.

### 16.5.3 *WASTE DISPOSAL STRATEGY*

Suitable mine waste rock, Ai and Bi categories, will be hauled from the pit and placed on two external waste disposal locations, North Dump and South Dump, both on the west side of the pit. Near the end of the mine life, there will be available space for an in-pit waste dump at the north end.

Waste material from the initial years and upper mining benches will be hauled to the South Dump. It will be constructed by a combination of staged lifts and wrap-arounds. Access from the pit will be from roads constructed along the contours at strategic elevations to maintain level or downhill hauls where possible. A wide berm will be constructed at the 1,040 m elevation to provide haulage and pipeline access to the tailings facility before the advancement of waste dump burying the initial road and corridor. The final elevation at the top of the waste dump will be 1,190 m.

The North Dump will be the waste rock disposal destination for mining benches at the north end of the pit, as well as from the lower elevations. This waste dump design is physically constrained by Spanish Creek to the north and Hepburn Lake to the northwest. The plant site is immediately to the south restricting its advancement in that direction. The top elevation will approximately be 1,075 m.

Development at the north end of the pit will be completed by Year 10 and will be available for backfilling. The majority of waste after Year 10 will be placed at this location.

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

Suitable mine waste rock, Ai and Bi categories, will also be hauled to the tailings facility for dam embankment construction as required. It is estimated that 36 Mt of rock will be required from the pit for both the north and south dam embankments through the LOM, including 7.2 Mt during the pre-production period. Limited samples from test pits indicate that the overburden is unconsolidated with high moisture content, and may not be competent for dam embankment construction. It is therefore assumed that only 50% of Ai overburden and 37% of the undefined category of overburden will be suitable for the dam.

Waste rock that requires management and cannot be placed in the waste dumps or used for construction of the tailings embankment will be placed in the tailings pond for subaqueous disposal. The ARD categories for this pit waste include Aii, Bii, and C. Some Aii and waste will be used for upstream tailings embankment construction in the pre-production period only, when the material will be submersed in less than two years.

## 16.6 Mine Plan Description

### 16.6.1 Pre-production Development

During the pre-production period 1 (PP 1) and pre-production period 2 (PP 2), mine related activities will include site clearing, stripping and stockpiling topsoil, establishing perimeter ditches, and haul road development. Approximately 11 km of haul roads will have to be constructed during the first pre-production period to access the top benches of Phase 1 from the ROM stockpile, and from the pit to the tailings facility and waste dump.

A total of 9.8 Mt of waste material will be pre-stripped to the 1,220 m bench of Phase 1 to ensure that mill feed of elevated head grades will be available at start-up. A total of 7.2 Mt of suitable waste rock will be hauled to the tailings embankment for construction commencing in the first pre-production period. A total of 0.2 Mt of waste will be hauled to the tailings pond for sub-aqueous disposal. Excess waste of 2.4 Mt will be hauled and placed on the South Dump while 1.9 Mt of mineralized material will be exposed and hauled to the ROM stockpile.

The top several benches in Phase 1 will be developed with a pioneering equipment fleet consisting of a small diameter track drill and dozers until a workable mining bench can be established for larger, more productive equipment. It is anticipated that this work will be carried out by a contract miner who will also construct the initial mine haulroads prior to the Owner's mine equipment fleet being available.

Figure 16.16 and Figure 16.17 illustrates the mining activities to be completed by end of PP 1 and PP 2, respectively.

**Figure 16.16 Mine Development – Pre-production Period 1 (PP 1)**

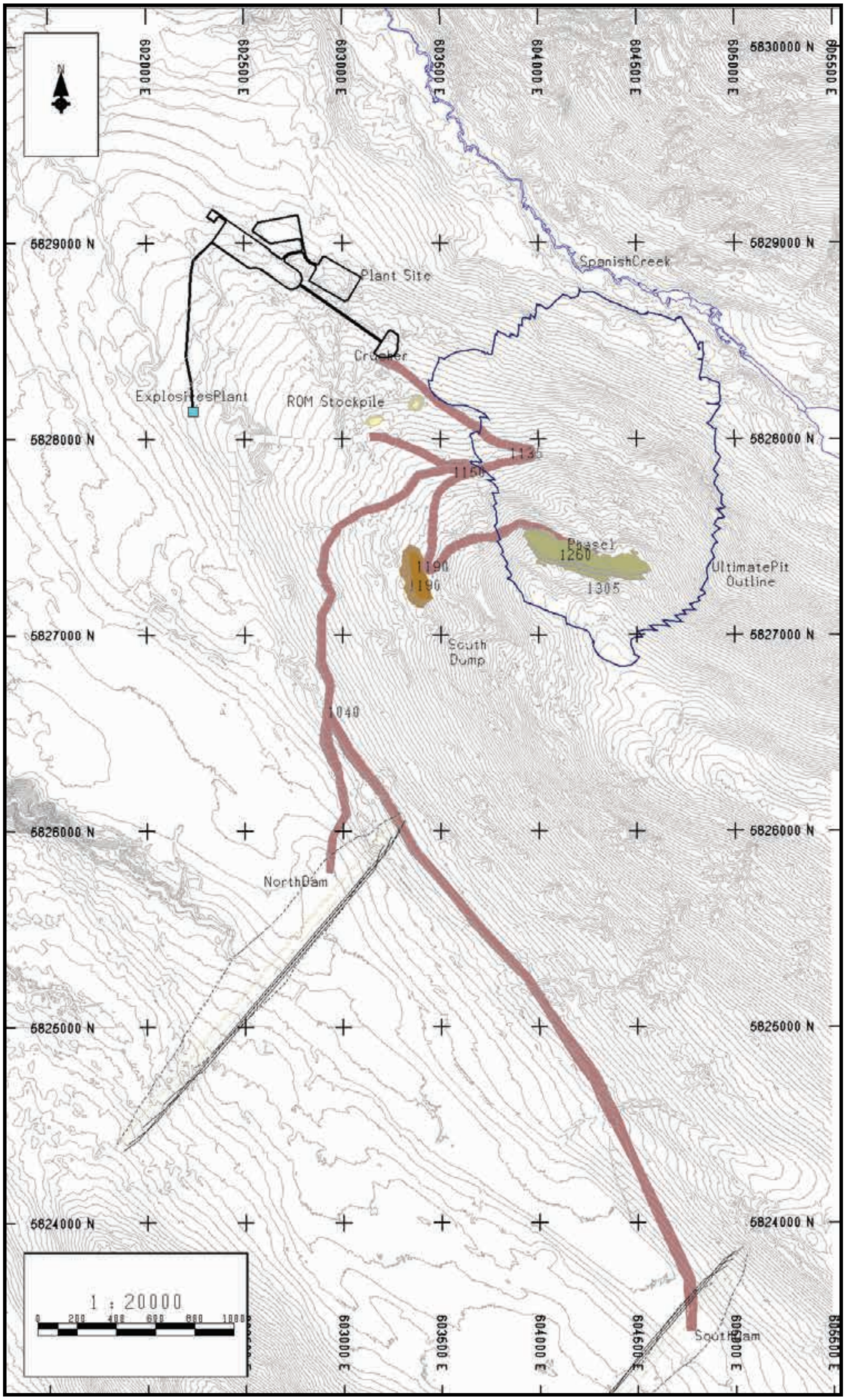
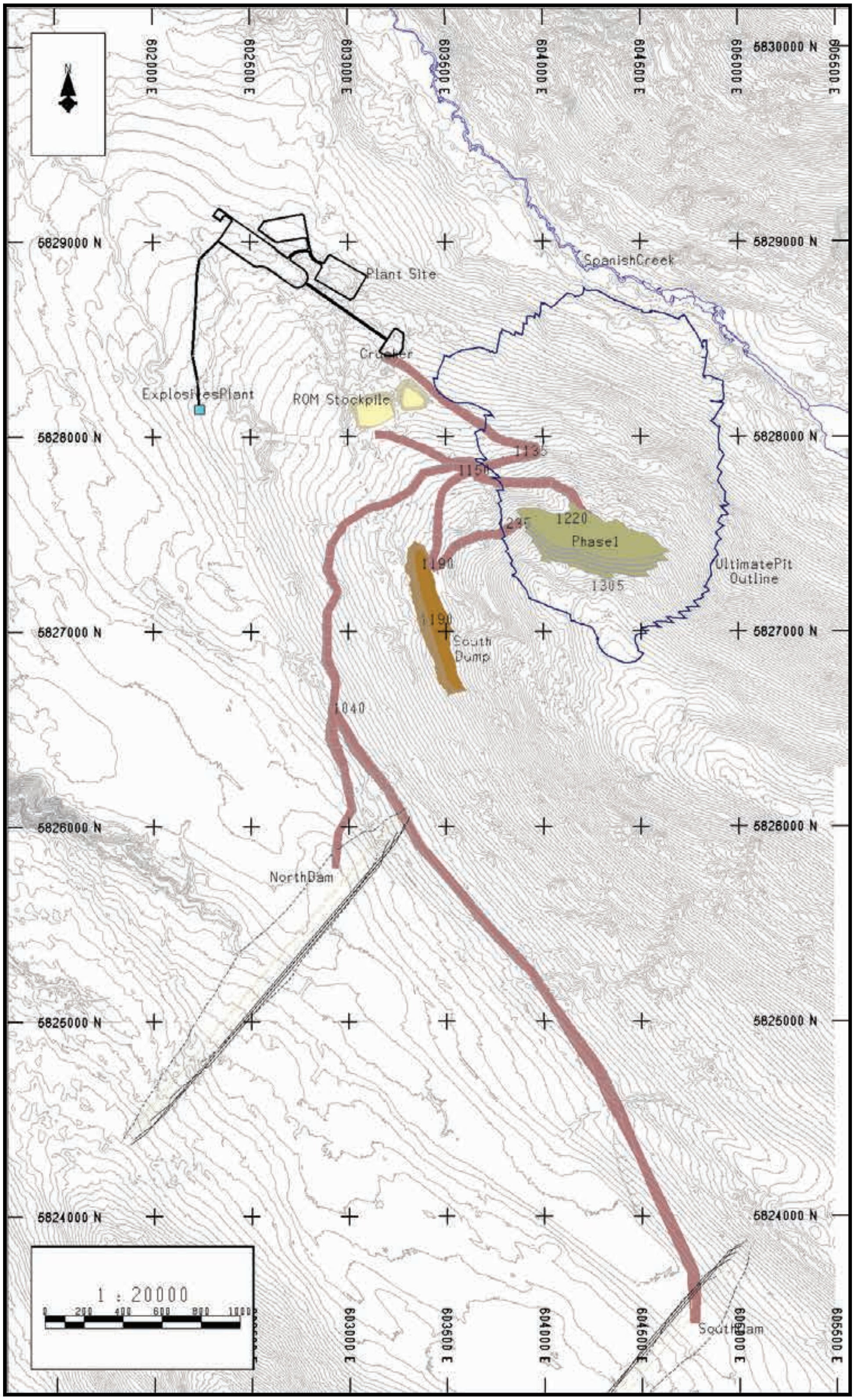


**Figure 16.17 Mine Development – Pre-production Period 2 (PP 2)**



### 16.6.2 General Mine Sequence

Mine production will commence in Phase 1 and will be the primary source for mill feed for the first three production years. Waste material will be hauled to the South Dump on lifts at various elevations leaving the pit. Where possible, haul roads will be level grades or downhill to minimize the haulage costs. Suitable waste material will continue to be hauled to the tailings facility for embankment construction.

Phase 2 will commence development in Year 2, and will also provide a source for mill feed by Year 3. Development of Phases 3 and 5 will follow in Years 3 and 4 to ensure continuous supply of mill feed when Phase 3 is completed in Year 4. Phase 4 is a small isolated pit on the west side that will be a minor source for mill feed from Years 6 to 11, with the majority in Year 6. Mining in Phase 6 will commence in Year 6, and development of accesses to the top at 1,360 m elevation will be necessary during the previous Year 5. The ridge along the top of Phase 6 is relatively gentle sloping and will not require much pioneering construction. Material mined will be hauled through Phase 5, and therefore it will be necessary to leave highwall road in Phase 5 as each bench is developed.

Figure 16.18 shows the development sequence for the pit phases.

The North Dump will be a waste destination by Year 4, as mining activities will be at the north end of the pit. This waste dump will store approximately 113 millions of waste material by the end of mining. Pit waste will be hauled annually to the tailings facility until end of Year 5. It will resume in Years 9 and 12 when the embankment will be at its final elevation.

By Year 5, South Dump will advance far enough westward to cover the haulroad and pipeline corridor to the tailings facility. An alternate corridor and road will be constructed in Year 4 on the 1,040 m elevation of the South Dump, and will replace those buried by the advancement of the waste dump.

The north end of the pit will be sufficiently completed by Year 10 to allow for waste backfilling. Approximately 73 Mt of waste material will be placed at this location.

Figure 16.19 to Figure 16.22 shows the mine development sequences for selected periods – Years 1, 6, 9, and 14. Year 14 is end of mining, with the only remaining activity being reclaiming the ROM stockpile that will occur in Year 15.

**Figure 16.18 Mine Sequence**

| | PP 1 | PP 2 | Y1 | Y2 | Y3 | Y4 | Y5 | Y6 | Y7 | Y8 | Y9 | Y10 | Y11 | Y12 | Y13 | Y14 | Y15 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Phase 1 (Starter Pit P616) | | | | | | | | | | | | | | | | | |
| Phase 2 (EW Pushback P626) | | | | | | | | | | | | | | | | | |
| Phase 3 (N Zone Pit P636) | | | | | | | | | | | | | | | | | |
| Phase 4 (West Pit P646) | | | | | | | | | | | | | | | | | |
| Phase 5 (S Pushback P656) | | | | | | | | | | | | | | | | | |
| Phase 6 (Final P666) | | | | | | | | | | | | | | | | | |

**Figure 16.19 Mine Plan – Year 1**

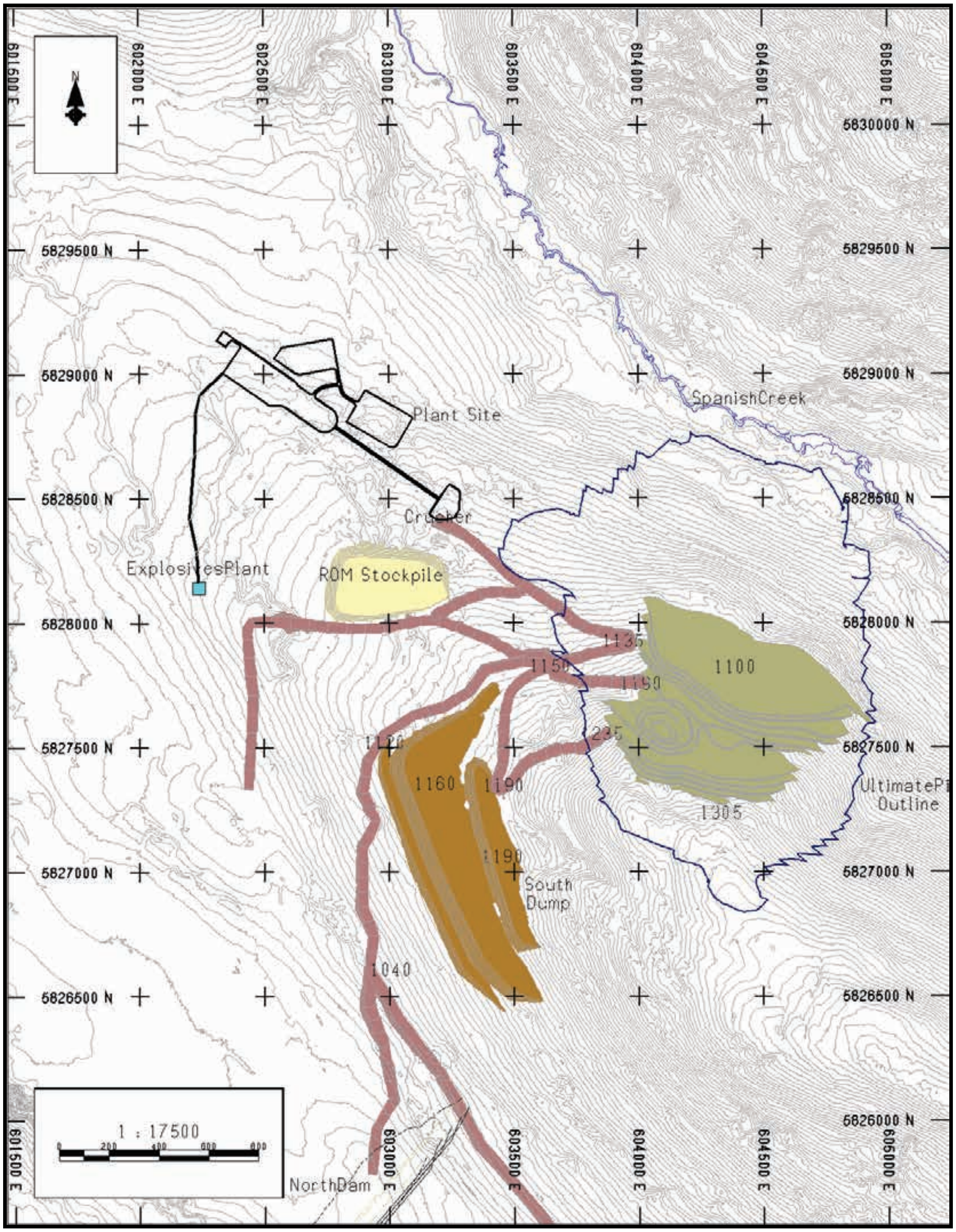


**Figure 16.20 Mine Plan – Year 6**

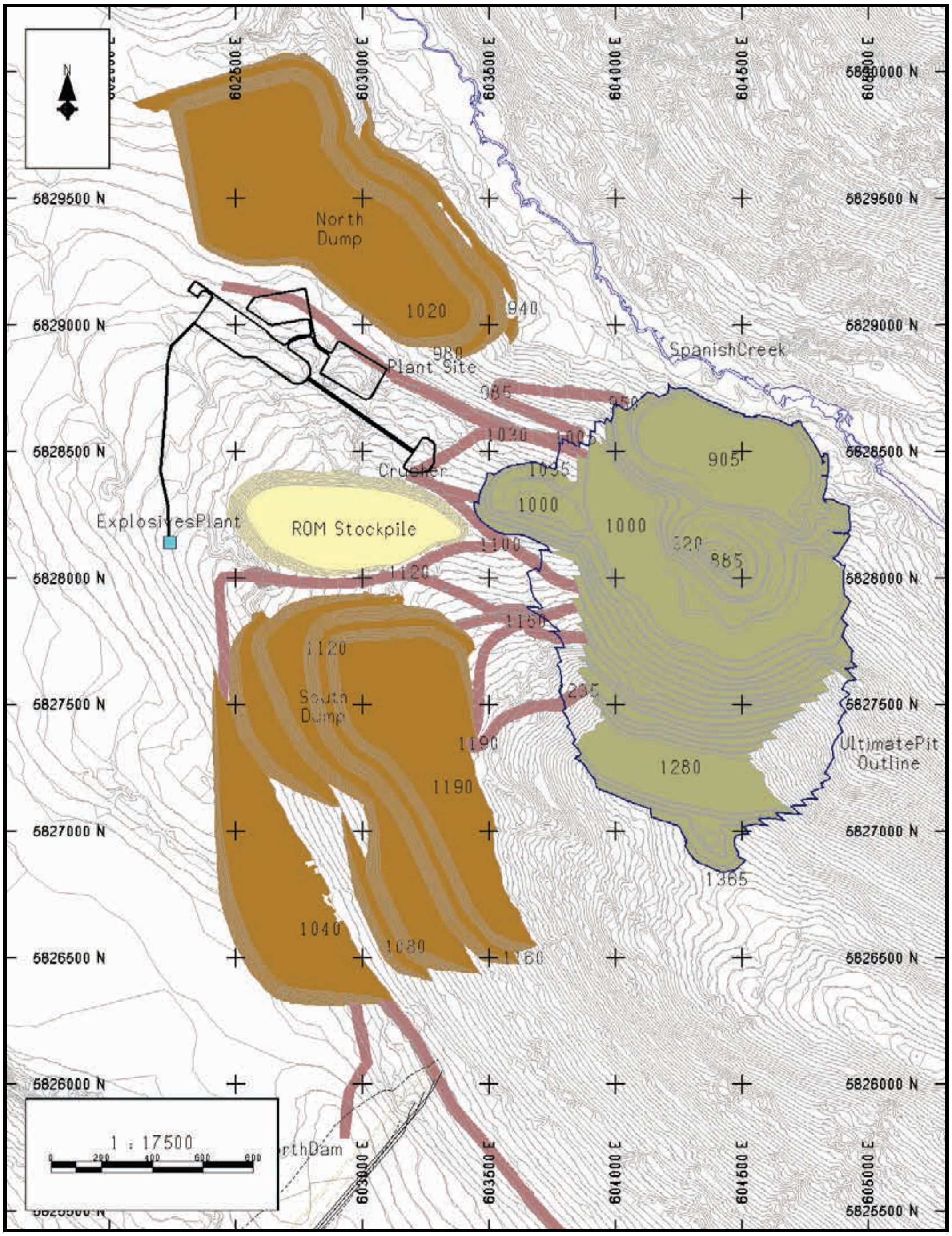


**Figure 16.21 Mine Plan – Year 9**

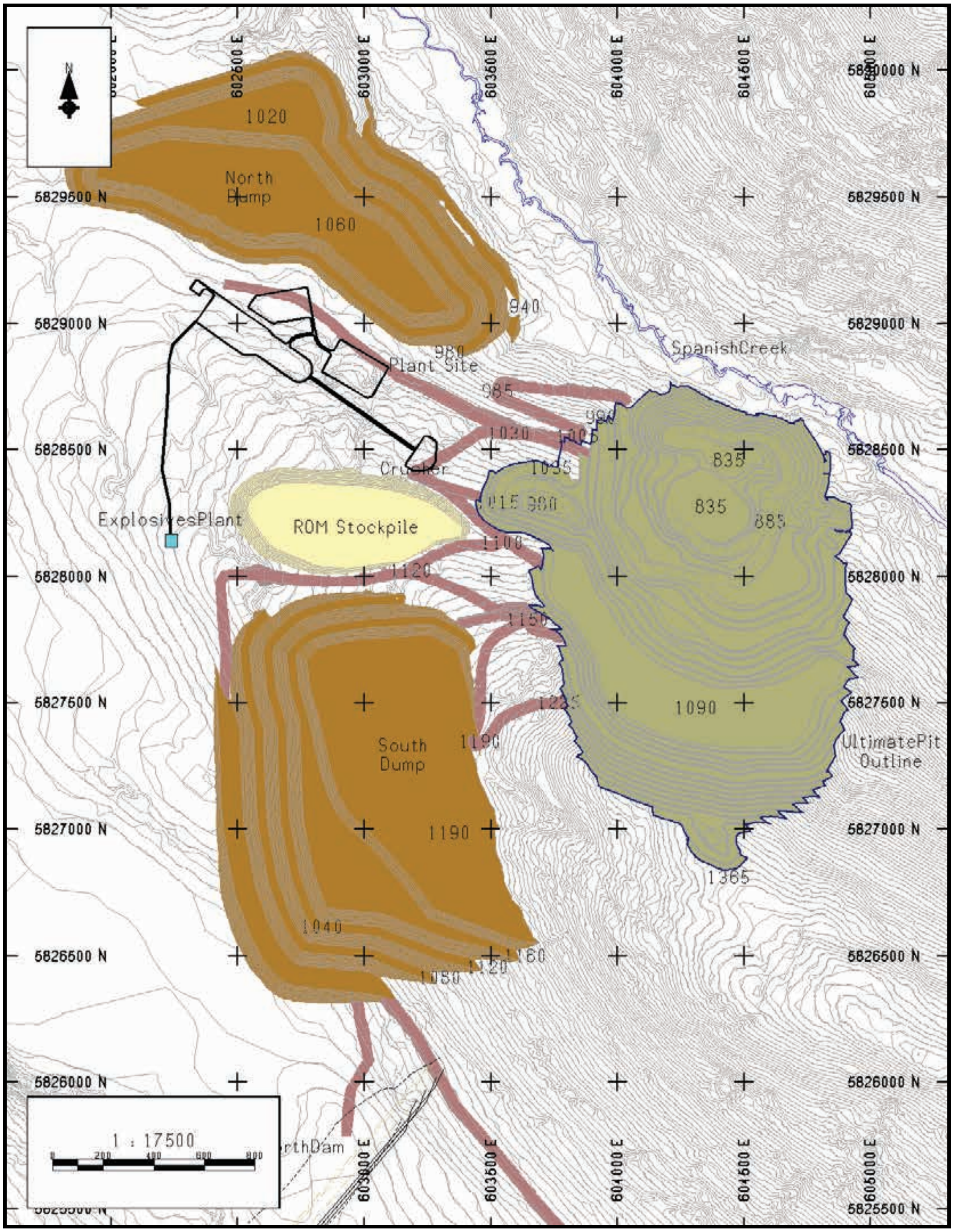
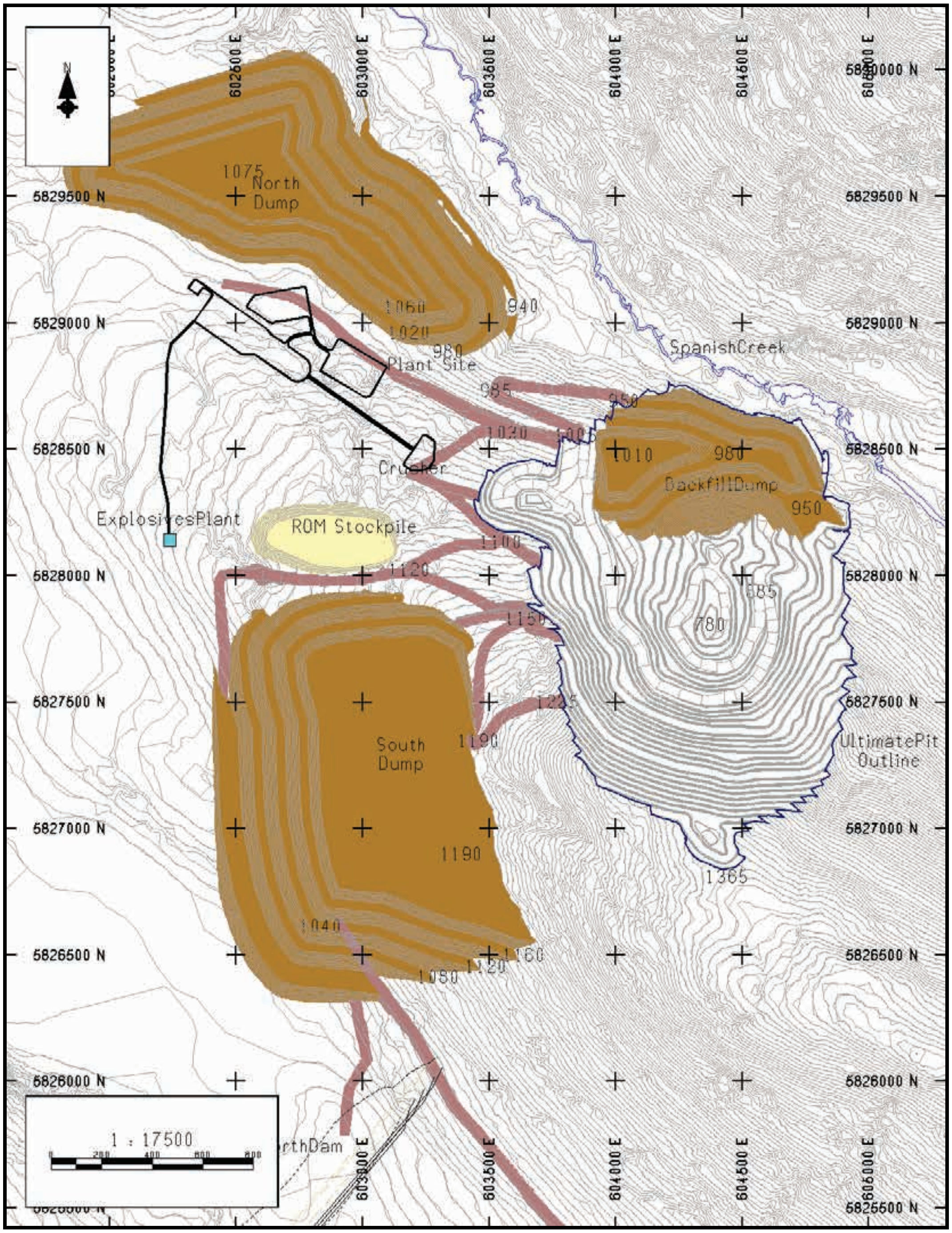


**Figure 16.22 Mine Plan – Year 14 (End of Pit Life)**



## 16.7 Mine Production Schedule

The mill feed rate will be 40,000 t/d. During the first production year, the average throughput will be reduced by one-third for start up inefficiencies.

Implementing a cut-off grade strategy is commonly utilized to optimize the mill feed so that the NPV of the Project is maximized. During the LOM, mineralized material mined at or above the gold grade value of 0.20 g/t will be recovered either by direct feed to the mill, or placed in the ROM stockpile. For the first three production years, elevated gold cut-off grades of 0.35, 0.30, and 0.25 g/t respectively, are applied to increase the mill feed grade. The material between 0.20 g/t and the elevated cut-off grade will be directed to the ROM stockpile and reclaimed in later years.

A preliminary production forecast is shown in Table 16.9. Over the life of the envisaged pit, approximately 188 Mt of mineralized material will be directly fed from the pit to the mill, and 18 Mt will be sent to the ROM stockpile. Most of the stockpile will be reclaimed at the end of mine life when the pit is depleted. The cumulative strip ratio (waste/mineralized material) for the first three years is 1.4:1. The strip ratio increases to an average of 3.7:1 during the periods from Years 6 to 10 when mining activities are in Phases 5 and 6, and the total material mined nears 70 Mt/a. As the LOM nears its end, the strip decreases to an average of 1:1 over the last four years.

**Table 16.9 Mine Production Forecast**

| | Mineralized Material | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| **Period** | **Pit to Mill (kt)** | **Pit to Stockpile (kt)** | **Stockpile to Mill (kt)** | **Total to Mill (kt)** | **Au Feed Grade (g/t)** | **Ag Feed Grade (g/t)** | **Waste Rock (kt)** | **Strip Ratio (t:t)** |
| PP 1 | - | 110 | - | - | - | - | 2,629 | 23.9 |
| PP 2 | - | 1,753 | - | - | - | - | 7,158 | 10.0 |
| 1 | 9,033 | 4,579 | 604 | 9,637 | 0.76 | 0.67 | 21,468 | 1.6 |
| 2 | 14,601 | 8,077 | - | 14,601 | 0.78 | 0.67 | 20,299 | 0.9 |
| 3 | 14,601 | 3,332 | - | 14,601 | 0.57 | 0.68 | 27,738 | 1.5 |
| 4 | 14,601 | - | - | 14,601 | 0.43 | 0.76 | 30,923 | 2.1 |
| 5 | 14,601 | - | - | 14,601 | 0.42 | 0.72 | 31,041 | 2.1 |
| 6 | 14,601 | - | - | 14,601 | 0.46 | 0.51 | 54,398 | 3.7 |
| 7 | 14,601 | - | - | 14,601 | 0.44 | 0.66 | 54,398 | 3.7 |
| 8 | 14,601 | - | - | 14,601 | 0.41 | 0.75 | 54,398 | 3.7 |
| 9 | 14,601 | - | - | 14,601 | 0.52 | 0.71 | 55,751 | 3.8 |
| 10 | 14,601 | - | - | 14,601 | 0.50 | 0.65 | 53,455 | 3.7 |
| 11 | 14,601 | - | - | 14,601 | 0.44 | 0.65 | 22,598 | 1.5 |
| 12 | 14,601 | - | - | 14,601 | 0.46 | 0.66 | 13,764 | 0.9 |
| 13 | 14,601 | - | - | 14,601 | 0.43 | 0.58 | 12,509 | 0.9 |
| 14 | 3,800 | - | 10,800 | 14,600 | 0.31 | 0.83 | 2,341 | 0.6 |
| 15 | - | - | 6,447 | 6,447 | 0.25 | 0.50 | - | - |
| **Total** | **188,045** | **17,851** | **17,851** | **205,896** | **0.48** | **0.67** | **464,870** | **2.3** |

Figure 16.23 and Figure 16.24 are graphical illustrations for the annual production and mill feed grades, respectively.

**Figure 16.23 Total Material Mined**

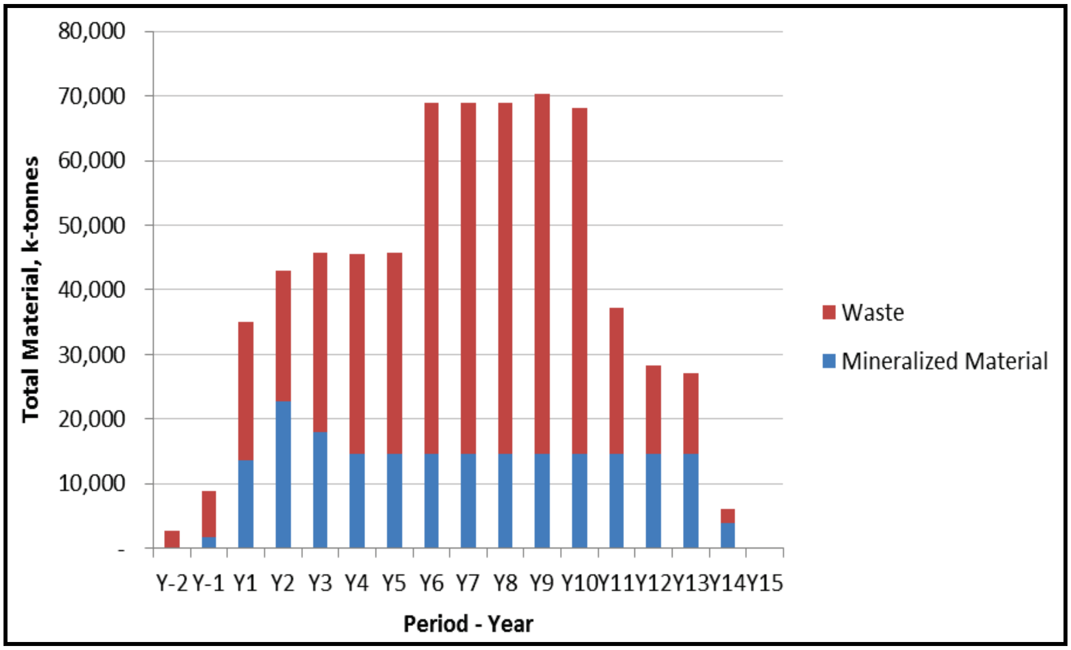
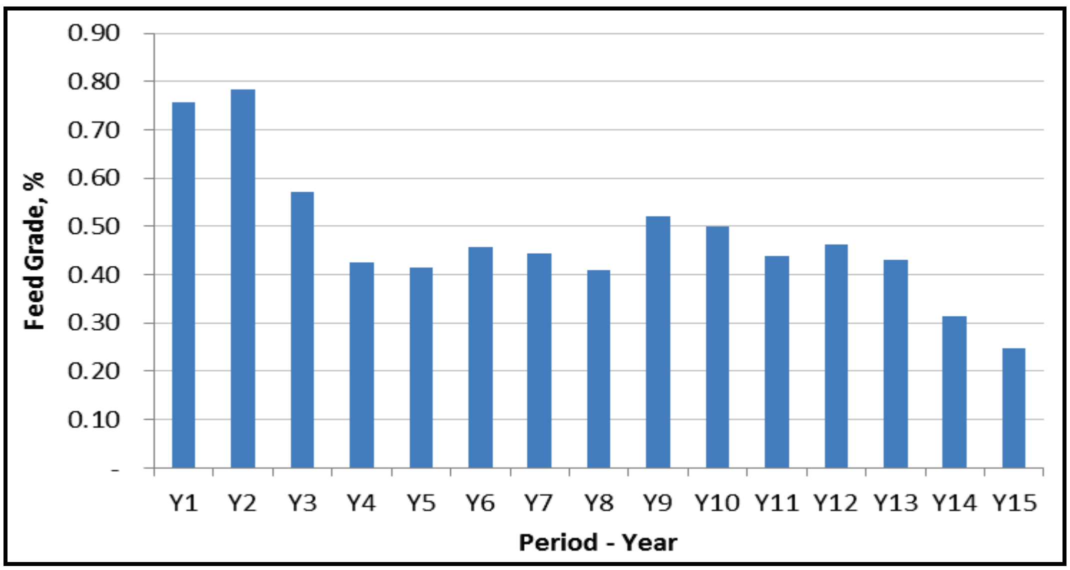



**Figure 16.24 Mill Feed Grade**




A schedule of the waste material quantities produced and their destinations are shown in Table 16.10. The waste dumps include a total of 207 Mt to the South Dump, 113 Mt to the North Dump, and 73 Mt backfilling the north end of the pit.

**Table 16.10 Mine Waste Disposal Quantities**

| Waste Quantity by Disposal Destinations (kt) | | | |
|---|---|---|---|
| | Dam Embankment | Sub-aqueous Disposal | Waste Dumps |
| PP 1 | 1,987 | 0 | 642 |
| PP 2 | 5,260 | 249 | 1,649 |
| 1 | 5,762 | 2,791 | 12,916 |
| 2 | 5,761 | 5,898 | 8,638 |
| 3 | 1,448 | 4,758 | 21,534 |
| 4 | 5,293 | 3,145 | 22,486 |
| 5 | 5,292 | 832 | 24,918 |
| 6 | - | 2,403 | 51,996 |
| 7 | - | 2,932 | 51,467 |
| 8 | - | 4,139 | 50,260 |
| 9 | 2,830 | 3,583 | 49,338 |
| 10 | - | 2,687 | 50,768 |
| 11 | - | 360 | 22,239 |
| 12 | 2,384 | 625 | 10,755 |
| 13 | - | 998 | 11,511 |
| 14 | - | 198 | 2,143 |
| 15 | - | - | - |
| **Total** | **36,017** | **35,598** | **393,260** |

## 16.8 Mine Equipment

### 16.8.1 Mobile Fleet

The mine fleet consists of the mobile equipment operating from the pit to the tipping point at the feed crusher, and to the waste disposal areas. It is assumed that the mine equipment fleet will be available on-site by Q2 of Year -2. Development work required prior to then will be undertaken by a contractor employing its own equipment fleet.

Table 16.11 summarizes the major equipment fleet, the number of units required at start-up, and the maximum fleet size during the LOM.

Table 16.11 Major Mine Equipment Fleets

| Major Mine Equipment | Purpose | Size | No. of Units | |
|---|---|---|---|---|
| | | | Y1 | Max. |
| Blasthole Diesel Drill | Primary Drill | 255 mm | 2 | 4 |
| Hydraulic Shovel | Production Loading | 21 $m^3$ | 3 | 5 |
| Wheel Loader | Backup Loader and Stockpile Handling | 18 $m^3$ | 1 | 1 |
| Haul Trucks | Production Haulage | 180 t | 10 | 25 |
| Track Dozer (D11 Equivalent) | Road Construction and Maintenance | 634 kW | 1 | 1 |
| Track Dozer (D10 Equivalent) | Waste Dumps and Road and Maintenance | 433 kW | 1 | 1 |
| Track Dozer (D9 Equivalent) | Pit Maintenance | 334 kW | 1 | 2 |
| Hydraulic Backhoe | Ditch Construction and Road Maintenance | 5 $m^3$ | 1 | 1 |
| | Pit Maintenance | 3 $m^3$ | 1 | 1 |
| Grader (16M Equivalent) | Road Maintenance | 4.9 m | 2 | 2 |
| Scraper | Topsoil/Overburden Removal | 345 kW | 1 | 1 |
| Water/Sanding Truck | Road Maintenance | 20,000 gal | 1 | 1 |

Equipment is selected based on the selective mining capability to minimize loss and dilution in the mill feed, while also achieving sufficiently high mining rates to ensure the lowest possible mine operating unit costs. It is anticipated that 20 to 25 $m^3$ sized hydraulic shovels with the capability to excavate in both front and backhoe configurations will meet these requirements. Mining benches will be 10 m high, and reduced to 5 m if a higher degree of mining selectivity is necessary. It is estimated that each unit will produce approximately 11 to 14 Mt of material annually depending on the loading conditions. Pit electrification will not be required as all equipment will be diesel powered.

The size of the trucks is selected to match the shovel output so that no more than four to five passes are necessary to fully load the truck. The size of the fleet is determined by estimating the haulage productivities for mineralized and waste materials. Preliminary estimates on the haulage productivities indicate that 10 units will be required by Year 1, increasing to a maximum of 25 units by Year 7.

The drill fleet will initially consist of two 255 mm diameter diesel blasthole drills for production drilling, increasing with scheduled material quantities to a maximum of four units by Year 6. For the purpose of this study, it is assumed that wall control will be established using buffer blasting techniques with the blasthole drill and a small diameter track drill will not be necessary. Further studies on rock structures and quality will determine whether pre-shearing with smaller diameter drillholes will be effective for highwall control.

Pit support equipment will include track dozers, and backhoes for pit floor maintenance, road development and maintenance, and ditching. The road maintenance fleet will also include motor graders and a water/sanding truck.

Ancillary mine equipment will include a small loader and truck fleet, fuel and lube truck, light duty vehicles, service trucks, cranes, utility backhoes, blasthole stemmers, lighting plants, in-pit pumps, and other equipment required to support the mine and maintenance areas of the operation.

### 16.8.2 MINE BUILDINGS

On-site mine service buildings will include a heavy-duty truck shop, mine dry, light duty vehicle shop, wash bay, warehouse/storage facility, fuel depot and distribution, assay laboratory facility, and administration-engineering offices.

Blasting explosives will be manufactured on-site, and the explosives plant will be housed in a secure structure. The plant and storage facilities will be located southeast of the plant site and pit, in compliance with regulatory requirements.

## 16.9 MINE LABOUR

The mine workforce estimate is summarized in Table 16.12. The hourly labour workforce reflects the material mined for during those periods. The staff positions are less variable with the mine production and are consistent throughout the LOM except for a reduction over the last five years. During the first five years the annual mine labour count averages 168. It increases to an average of 228 during the next five year period when the average waste production rate more than doubles. When the strip ratio is reduced after ten years, the mine labour force is reduced significantly.

**Table 16.12 Mine Operations Labour – Average for Periods**

| | Year | | |
|---|---|---|---|
| | **1 to 5** | **6 to 10** | **11 to 14** |
| Equipment Operators | 102 | 146 | 70 |
| Mine Maintenance | 35 | 51 | 25 |
| **Subtotal** | **137** | **197** | **95** |
| Mine Superintendent | 1 | 1 | 1 |
| Mine & Maintenance General Foremen | 2 | 2 | 1.5 |
| Shift Foremen/Team Leaders | 9 | 9 | 7 |
| Trainers | 1 | 1 | 0.5 |
| Maintenance Planners | 1 | 1 | 0.7 |
| Clerks | 2 | 2 | 1.5 |
| **Subtotal** | **16** | **16** | **12.2** |
| **Mine Technical** | | | |
| Chief Engineer | 1 | 1 | 1 |
| Geologists | 3 | 3 | 2 |
| Mine Engineers | 6 | 6 | 4 |

*table continues…*

| | Year | | |
|---|---|---|---|
| | 1 to 5 | 6 to 10 | 11 to 14 |
| Technicians/Surveyors | 5 | 5 | 4 |
| **Sub-total** | **15** | **15** | **11** |
| **Total Salaried Staff** | **31** | **31** | **23** |
| **Total Mine Workforce** | **168** | **228** | **118** |

# 17.0 RECOVERY METHODS

## 17.1 INTRODUCTION

The unit processes selected for the design of the process plant were based on the results of metallurgical testing performed at G&T and SGS, along with the Project-related parameters set out by SMG. The metallurgical processing procedures selected for the design will produce gold-silver doré as a final product.

The 40,000 t/d process plant flowsheet design follows conventional crushing and a semi-autogenous mill with pebble crushing and ball mill grinding circuit (semi-autogenous-ball milling-crushing (SABC)) with cyclone classification. A gravity concentration circuit that uses a centrifugal concentrator will be included in the grinding circuit for the recovery of liberated gold. The cyclone overflow will report to the flotation circuit to produce a sulphide mineral concentrate containing the precious metals. The flotation circuit will incorporate a pre-flotation stage followed by a rougher stage with two open-circuit cleaner stages. A scavenging gravity concentration circuit will also be included within the flotation circuit, in order to limit gold losses from the open flotation circuit configuration. Flotation tailings together with the gravity scavenger tailings will be directed to the tailings impoundment area for storage.

There will be two leaching circuits, namely: an intensive cyanide leach circuit which will treat the primary gravity concentrate, and a CIL circuit which will treat the flotation concentrate. Prior to the CIL circuit, the flotation concentrate will be thickened and finely ground to enhance leaching kinetics. The reground concentrate will be pre-aerated and then treated in the CIL circuit to recover gold from the feed material. Loaded carbon will be transferred from the head CIL tank to the elution circuit on a daily basis, while regenerated and/or fresh carbon will be brought from the carbon circuit for maintaining the carbon concentration in the CIL circuit. The loaded carbon will initially be acid-washed to remove calcium and other impurities to be followed by the gold elution, or stripping, process. The gold will be recovered from the elution solution, or pregnant solution, by electrowinning. The eluted carbon will be regenerated in a kiln prior to screening for the removal of carbon fines. The screened regenerated carbon will subsequently be returned to the adsorption circuit. The CIL tailings will be pumped to the cyanide detoxification tank where cyanide levels will be chemically reduced to acceptable environmental levels prior to disposal to the TSF.

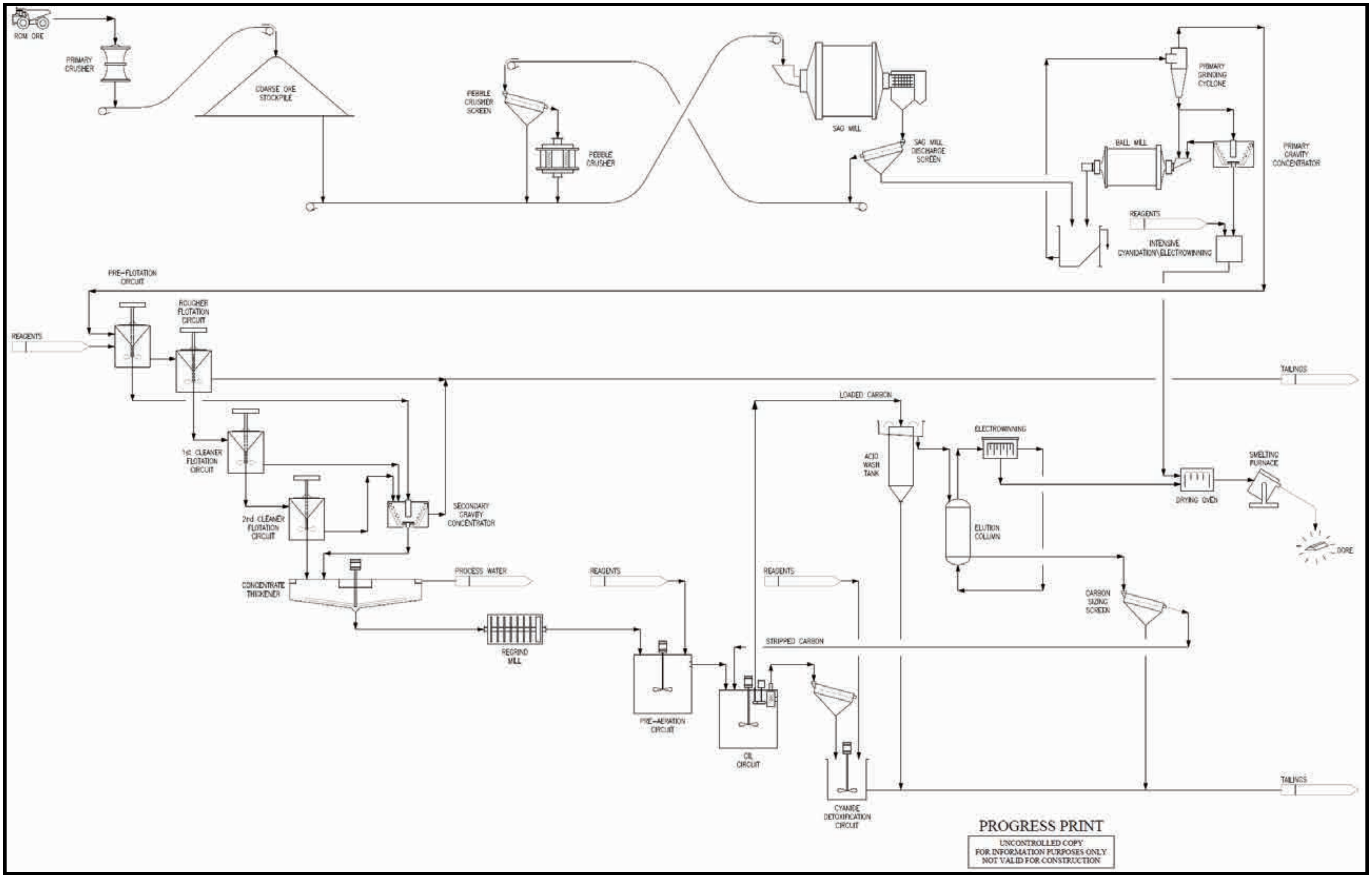
Process water will be recycled from the flotation concentrate thickener overflow and then supplemented with process water recovered from the tailings impoundment facility. Fresh water will be used for gland service and reagent preparation while treated process water will be used in the gravity circuit for fluidisation.

The 40,000 t/d process plant will consist of the following unit operations and facilities:

- ROM material receiving and primary crushing
- coarse plant feed material stockpile
- coarse plant feed material stockpile reclaim facilities
- a SAG mill and ball mill grinding circuit incorporating cyclones for classification
- SAG mill pebble crushing circuit
- centrifugal gravity concentration recovery circuit with intensive cyanidation leach circuit and electrowinning
- pre-flotation stage
- sulphide mineral rougher flotation
- two-stage sulphide mineral flotation cleaner circuit
- scavenging centrifugal gravity concentration circuit
- flotation concentrate regrinding stage
- concentrate thickening
- pre-aeration circuit
- CIL cyanide leaching and carbon adsorption circuit
- carbon handling and treatment
- electrowinning and smelting (gold refining)
- CIL tailings cyanide detoxification
- tailings deposition
- process water reclamation.

The simplified flowsheet is shown in Figure 17.1.

**Figure 17.1 Simplified Process Flowsheet for 40,000 t/d**



## 17.2 Plant Design

### 17.2.1 *Major Design Criteria*

The concentrator has been designed to treat gold-bearing material at the rate of 40,000 t/d, (14,600,000 t/a). The major design criteria are outlined in Table 17.1. The complete design criteria are located in Appendix F. The process plant material balance and water balance are located in Appendix G.

**Table 17.1 Major Design Criteria**

| Criteria | Unit | Value |
|---|---|---|
| Overall Plant Feed | t/d | 40,000 |
| Operating Year | d | 365 |
| Primary Crushing Circuit Utilization | % | 80 |
| Grinding, CIL and Carbon Circuits Utilization | % | 92 |
| Primary Crushing Circuit Throughput Rate | t/h | 2,083 |
| Grinding and Flotation Process Rate | t/h | 1,812 |
| Leaching Process Rate | t/h | 52.9 |
| Bond Ball Mill Work Index, design | kWh/t | 13.3 |
| Bond Abrasion Index, Design | g | 0.269 |
| Specific Gravity Feed | - | 2.76 |
| Moisture Content Feed | % | 5.0 |
| SAG Mill Feed Size, 80% Passing | µm | 150,000 |
| Ball Mill Feed Size, 80% Passing | µm | 1,750 |
| Ball Mill Product Size, 80% Passing | µm | 184 |
| Ball Mill Circulating Load | % | 250 |
| Regrind Mill Product Size, 80% Passing | µm | 20 |
| Pre-aeration Time | h | 12 |
| Leach Circuit Retention Time | h | 53 |
| Head Grade Years 1-3, Design | Au, g/t | 0.70 |
| Head Grade Year 4 Onwards | Au, g/t | 0.43 |
| Head Grade Years 1-3, Design | Ag, g/t | 0.67 |
| Head Grade Year 4 Onwards | Ag, g/t | 0.67 |
| Anticipated Recovery Years 1-3, Design | Au, % | 90.3 |
| Anticipated Recovery Year 4 Onwards | Au, % | 87.0 |
| Anticipated Recovery, Design | Ag, % | 25.0 |

The design parameters are based on test work results obtained by G&T, particularly from test programs performed in 2011 and 2012. Data from the SGS's test report completed during 2010 was also incorporated in the design.

The grinding mills were sized based on the Bond work index data obtained from the testwork. The regrind mills were sized with assistance from the vendor.

The flotation cells were sized based on the optimum flotation times as determined during the laboratory flotation test work program. Typical flotation cell design parameters have been used in the design of the flotation circuit.

The leach circuit was sized based on the optimum leach retention times as determined during the laboratory leaching tests. Typical plant design parameters have been used in the design of the CIL and carbon treatment and handling circuit.

### 17.2.2 Operating Schedule and Availability

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

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

## 17.3 Process Plant Description

### 17.3.1 Primary Crushing

The crushing circuit will reduce the mined material from a nominal top size of 1,000 mm to a product size $P_{80}$ of 150 mm in preparation for the grinding process. The primary crushing circuit will contain the following main items of equipment:

- ROM feed hopper/surge bin
- oversize rock breaker
- gyratory crusher, Superior MKII 54-75 or equivalent
- conveyor belts
- belt scale
- dust collection systems.

Haul trucks with a capacity of 180 t will bring ROM plant feed material to the dry crushing plant. The material will be dumped directly from the trucks into the feed hopper for crushing. The feed hopper will have a surge capacity of just over one truck-load. The surge bin will be equipped with a coarse rock breaker in the event that oversize material makes its way into the feed hopper. ROM material will exit the feed hopper and will be delivered to the primary gyratory crusher. The gyratory crusher will operate at a utilization factor of 80% and will process 2,083 t/h of material. The gyratory crusher will reduce the size from 1,000 mm to a product size $P_{80}$ of about 150 mm with a closed side setting of 165 mm. The crushed material will then be deposited onto the conveyor belt feeding the crushed material stockpile.

The primary crushing and conveyor drop points will be equipped with a dust collection system to control fugitive dust that will be generated during crushing and during conveyor loading and the transportation of the crushed material.

### 17.3.2 Crushed Material Stockpile and Reclaim

The crushed material stockpile will be a production surge facility which will allow for a steady feed to the grinding and flotation circuits.

The major equipment and facilities in this area includes:

- crushed material stockpile, 40,000 t live capacity
- reclaim belt feeders
- conveyor belts, metal detectors, self-cleaning magnets, and belt tear detectors
- belt scale
- dust collection system.

The coarse crushed material stockpile will have a live capacity of 40,000 t. The material will be reclaimed from this stockpile by belt feeders at a nominal rate of 1,812 t/h. The belt feeders will feed the reclaim conveyor which in turn will feed the SAG mill. The SAG mill feed conveyor will be equipped with a belt scale.

The coarse material stockpile and reclaim area will be equipped with a dust collection system to control fugitive dust that will be generated during conveyor loading and the transportation of the feed material.

### 17.3.3 Grinding Circuit Operation

The grinding circuit will reduce the size of the crushed material to a final product size of $P_{80}$ of 184 µm, which is suitable for gold recovery by gravity concentration, flotation and CIL. The grinding process will be a two-stage operation with the SAG mill in closed circuit with a pebble crusher, and the ball mill in closed circuit with the classifying cyclones. The SAG mill will be equipped with pebble ports to remove pebbles coarser than 65 mm. The grinding will be conducted as a wet process at a nominal rate of 1,812 t/h of material. The grinding circuit will include the following main items of equipment:

- SAG mill – 11.0 m diameter by 5.5 m long (36 ft by 18 ft), 11,500 kW installed power

- ball mill – 7.9 m diameter by 12.3 m long (26 ft by 40 ft), 14,500 kW installed power
- mill discharge pumpbox
- cyclone feed slurry pumps
- classification cyclone cluster
- vibrating trash screen
- mass flow meter
- particle size analyzer
- plant feed sampler system.

The rock material from the crushed material stockpile will be reclaimed under controlled feed rate conditions using reclaim belt feeders. These feeders will discharge the material onto the reclaim conveyor belt which will feed onto the SAG mill feed conveyor belt. A belt scale will control the feed to the SAG mill. Water will be added to the SAG mill feed material to assist the grinding process as required, and to maintain the slurry density at 72% solids. The SAG mill will operate at a critical speed of 70%.

The SAG mill discharge end will be equipped with 65 mm pebble ports to remove the critical size material. The mill discharge will be screened by the mill trommel and SAG mill screen. The oversize material will be conveyed via transfer conveyors to the pebble crushers. The cone crushers will crush the pebbles to a $P_{80}$ of 15 mm. The crushed material will be returned to the conveyor belt feeding the SAG mill for further grinding. The trommel underflow will be discharged into the mill discharge pumpbox.

In the mill discharge pumpbox, the SAG mill discharge will combine with the ball mill discharge to become the feed to the classification cyclones. The classification circuit will consist of one cyclone cluster. The slurry in the mill discharge pumpbox will be pumped to the cyclone cluster for classification. The cut size for the cyclones will be at a particle size $P_{80}$ of 184 µm, and the circulating load to the ball mill will be 250%. The cyclone underflow will be returned to the ball mill as feed material. A 48% portion of the cyclone underflow stream will be directed to the gravity concentration circuit for gold recovery processing. The ball mill will operate at a critical speed of 75%. Feed to the ball mill will consist of a portion of the cyclone underflow stream together with the gravity circuit tailings.

The mill feed rate of 1,811 t/h of new feed will constitute the feed to the flotation circuit. The overflow from the classification circuit will initially feed a vibrating trash screen before entering the flotation circuit. Process water will be used to spray the deck of the screens to wash off any adhering material from the oversize tramp material or trash, which will be trash collected in totes and emptied as required.

The trash screen underflow will be discharged by gravity into the first stage of flotation, namely the pre-flotation circuit. The pulp density of the screen underflow slurry will be approximately 37% solids.

Grinding media will be added to both the SAG and ball mills in order to maintain the grinding efficiency. The steel balls will be added periodically using a ball charging kibble.

#### 17.3.4 Gravity Concentration Including Intensive Cyanidation and Electrowinning Operations

The gravity concentration circuit will produce a concentrate from the grinding circuit containing gravity recoverable gold. The gravity concentrate will be treated in a dedicated intensive cyanide leach and electrowinning circuit to recover the gold from the concentrate and resulting pregnant solution prior to smelting the cathode gold at the refinery.

The main items of equipment in this circuit will be the following:

- gravity circuit feed preparation sizing screen
- gravity concentrator feed distributor
- centrifugal gravity concentrators, eight KC-QS48 units, or equivalent
- concentrate holding tank
- intensive cyanide leach module, CS6000, or equivalent
- dedicated electrowinning circuit.

A nominal 53% portion, equivalent to 2,400 t/h, of the cyclone underflow from the grinding circuit will be re-directed to the gravity circuit as gravity circuit feed. The gravity circuit feed will initially be screened over a vibrating screen in order to remove oversize and grit particles which are greater than 2 mm in size. The screen oversize material will be returned to the grinding circuit for further grinding. The screen undersize will enter the gravity concentrator feed distributor which will split the stream eight ways to feed the individual centrifugal concentrators.

The concentrators will operate continuously on a batch basis, and will flush once every hour to remove the concentrate collected in the unit. The concentrator flush material will be deposited into the concentrate holding tank, which will collect all the concentrate from the centrifugal concentrators over a whole day of production, an amount of approximately 8.6 t. Gravity concentrator tailings will be discharged from the concentrators and returned to the grinding circuit.

The gravity concentrate will be stored until a sufficient amount has been collected for the intensive cyanidation process. Intensive cyanidation of the gravity concentrate will normally be conducted as a batch process on a daily basis. The concentrate will initially be elutriated with water in order to remove any accompanying slimes. The excess water will be decanted and returned to the grinding circuit. The solution will initially be made up to a pH value of 10.5 to 11.0 with caustic solution. Cyanide will then be added to the solution in the reactor feed vessel. The cyanide concentration will be 1,000 g/L sodium cyanide. This solution will then be heated to a temperature of about 50°C. A leach accelerant, such as hydrogen peroxide, or leach-aid, will also be added to the cyanide solution to provide oxygen and to increase the leach kinetics. The solution will be pumped through the bottom of the reactor tank forming a fluidized bed of the gravity concentrate in the tank thereby maximizing the contact of the concentrate with cyanide solution. The solution will overflow the reactor tank and will be returned to the reactor feed tank where the pH and the cyanide concentration will be adjusted, if necessary, before pumping the solution back into the reactor tank. When the cyanidation process has been completed, the now gold-bearing pregnant solution will be transferred to the dedicated electrowinning circuit for gold and silver recovery. The leach residue will be flushed out of the reactor tank and returned to the process as feed to the CIL feed thickener. The duration of the intensive cyanidation leach process cycle is expected to be between 12 and 16 hours. The amount of gravity concentrate which will be treated on a daily basis is estimated to be 8.6 t although the reactor tank will have a nominal capacity of 9.0 t per batch of concentrate that can be treated.

The pregnant cyanide solution from the leaching process will be transferred to the dedicated electrowinning circuit where the precious metals will be recovered. The barren solution will be re-used within the intensive cyanidation circuit while the electrowinning sludge will be taken to the refinery to be dried and transferred to the furnace for smelting together with the gold production from the CIL circuit.

### 17.3.5 *Flotation Circuit*

The milled pulp will be subjected to flotation to recover the targeted minerals into a gold-silver bearing concentrate. Tank type flotation cells will be used throughout the flotation circuit.

The process objective will be to recover the gold and associated silver in a flotation concentrate while minimizing the amount of preg-robbing and gangue minerals present. Process selection was based on the nature of the material to be treated and included the use of a pre-flotation stage and rougher flotation with no regrinding of the concentrate within the flotation circuit, nor the use of recycle streams. The process design does incorporate a scavenging gravity concentration step to minimize the potential loss of gold.

The rougher concentrate stream will constitute the feed for the cleaner flotation sections of the flotation circuit. The circuit will consist of two stages of cleaning with no recycle streams.

The flotation circuit will include the following equipment:

- flotation reagent addition facilities

- pre-flotation tank cells, 2 x 300 $m^3$ and 300 kW installed power,
- conditioning tank, 5.0 m diameter x 5.6 m
- rougher flotation tank cells, 6 x 300 $m^3$ and 300 kW installed power per cell
- first cleaner flotation tank cells, 4 x 30 $m^3$ and 45 kW installed power per cell
- second cleaner flotation tank cells, 3 x 30 $m^3$ and 45 kW installed power per cell
- pumpboxes
- air blowers
- slurry and concentrate pumps
- sampling facilities.

The cyclone overflow from the grinding circuit will feed the flotation circuit by gravity flow from the ball mill grinding circuit cyclone cluster. The slurry will be monitored for particle $P_{80}$ size, and flotation feed samples will be taken periodically for process control and metallurgical accounting.

The cyclone overflow from the grinding circuit will discharge into the feed end of the pre-flotation series of tank cells. The slurry will enter the circuit at the design feed rate of 1,811 t/h. Air will be supplied to all the flotation cells. Only the frother, MIBC, will be added to the pre-flotation stage. The purpose of pre-flotation will be to remove carbonaceous material from the circuit prior to the downstream sulphide flotation circuit and CIL process. The pre-flotation concentrate will constitute approximately 5% mass of the plant feed. The pre-flotation concentrate will gravity flow to the feed tank of the secondary gravity circuit. The pre-flotation circuit tailings will form the feed for the rougher flotation circuit.

The pre-flotation tailings will enter the rougher circuit conditioning tank which , which has been sized for a retention time, or conditioning period, of four minutes. The amount of flotation reagents (i.e. collector PAX, the frother MIBC, and the depressant CMC) which will be added to the conditioning tank will be as defined through testing. Provision will be made in the design for the staged addition of the reagents in the rougher circuit as well as in the cleaner stages of the flotation circuit.

The rougher feed slurry will overflow the conditioning tank into the feed end of the rougher flotation tank cell line. The bulk flotation of the sulphide minerals together with the associated gold and silver will take place in the rougher flotation circuit. The rougher concentrate will constitute approximately 6% mass of the plant feed. The rougher tailings will be sampled automatically prior to discharge into the flotation tailings pumpbox for process control and metallurgical accounting purposes. This tailings stream will constitute one of the final tailings streams leaving the plant.

The rougher flotation concentrate will be the feed to the first cleaner flotation stage. Tailings from the first cleaner flotation stage will report to the secondary gravity concentration surge tank. The first cleaner concentrate will constitute the feed to second cleaner flotation stage. Tailings from the second cleaner flotation stage will also report to the secondary gravity concentration surge tank. The second cleaner concentrate will be the final concentrate. The final concentrate will comprise approximately 3% of the original plant feed mass, expected to be 52.5 t/h. The final flotation concentrate will be delivered to the CIL feed thickener for dewatering.

The concentrate thickener overflow water will be re-used in the grinding and flotation circuit as process water, provided it will not harm the flotation process.

### 17.3.6 SECONDARY GRAVITY CONCENTRATION CIRCUIT

The secondary gravity concentration circuit will be the gold scavenging circuit, which will mitigate against possible gold recovery losses from the flotation circuit. Gold recovery losses could occur within the flotation circuit because the flotation circuit design does not incorporate typical recycle streams. The removal of the recycle streams will potentially minimize the build-up of gangue and the consequent accumulation of carbonaceous material being fed into the CIL circuit with the flotation concentrate.

The pre-flotation concentrate, first cleaner tailings and second cleaner tailings will constitute the feed to the secondary gravity concentration circuit. This is an unusual application and it is based on recent changes in industry in the recovery of fine gravity gold. Typically, the gravity circuit would be installed in the cyclone underflow of a grinding circuit and would only treat a portion of the cyclone underflow. In this application, the centrifugal concentrator will be acting as a gold scavenger and gangue/carbon rejecter.

The secondary gravity concentration circuit feed material will be pumped to the centrifugal concentrator. This gravity circuit will also be equipped with a trash screen and a centrifugal gravity concentrator. The gravity concentrator will recover any residual particles of gold and high density gold-bearing sulphide mineral particles. The gravity tailings will be discharged into the secondary gravity concentration tailings pumpbox which will allow the material to be pumped to the tailings dam feed pumpbox. A sample of this tailings stream will be collected prior to disposal for metallurgical accounting purposes. The collected gravity concentrate will periodically be flushed into the secondary gravity concentrate holding tank. Each flush cycle will generate approximately 45 kg of concentrate. This tank will act as a collector tank for the gravity concentrate as well as control the intermittent transfer of the gravity concentrate as feed to the CIL feed thickener which will be the final destination for the secondary gravity concentration concentrate.

The centrifugal concentrator will be equipped with an automated concentrate purging programmable logic controller (PLC) which will minimize operator intervention.

### 17.3.7 CIL Circuit: Feed Thickening

The cleaner flotation concentrate, the secondary gravity circuit concentrate and the primary gravity circuit intensive cyanidation tailings, will be combined and prepared for leaching by increasing the pulp density in the CIL feed thickener prior to this material entering the regrinding stage. The thickening circuit will have the following equipment:

- CIL feed thickener, 20.0 m diameter
- CIL feed thickener overflow water standpipe
- CIL feed thickener overflow water pumps
- CIL feed thickener underflow slurry pumps.

The cleaner flotation concentrate will be pumped from the final cleaner flotation stage to the concentrate thickener, where it will be combined with the secondary gravity concentrate in the thickener feed well. Flocculant will be added to the thickener feed to aid the settling process. The thickened concentrate (now known as the CIL feed) will be pumped to the regrind IsaMill™ trash screen at the head of the regrind circuit. The regrind mill will reduce the particle size of the material to about 20 µm prior to cyanidation in the CIL circuit. The thickener underflow slurry density will be 50% solids.

The CIL feed thickener overflow solution will be collected in the process water tank for recycling within the mill circuit.

A mass flow meter will monitor the process feed rates.

### 17.3.8 CIL Circuit: IsaMill™ Regrind Stage

To enhance leach kinetics, the regrind mill will be used to reduce the particle size of the CIL feed to a $P_{80}$ size of 20 µm in order to liberate and expose the finer gold particles and silver-bearing minerals associated with, and/or locked within sulphide and gangue minerals. The main items of equipment will be the following:

- vibrating trash screen
- IsaMill™ M5000 regrind mill, 5,000 L capacity and 1,500 kW installed power
- grinding media addition system
- pumpboxes
- slurry pumps.

The CIL feed thickener underflow, which will be the feed material for the regrind circuit, will pass through the trash screen and enter the IsaMill™ feed pumpbox. Since the regrind feed has been thickened, there is no need for cyclone densification within the regrind circuit.

The thickener underflow will be pumped from the feed pumpbox into the IsaMill™ where it will be ground to the final design particle $P_{80}$ size of 20 µm. The finely ground product will discharge into the discharge pumpbox from where it will be pumped to the CIL pre-aeration tank at the head of the leach circuit. The regrind IsaMill™ circuit will have a design treatment rate of 53 t/h and will have a circulating load of approximately 10% around the mill as specified by the vendor.

### 17.3.9 CIL Circuit: Pre-aeration

The thickened reground flotation concentrate will be aerated with oxygen under alkaline conditions in order to facilitate the cyanide leaching process. The main items of equipment in this section will be:

- CIL feed distributor
- CIL pre-aeration tank, 10.0 m diameter by 10.6 m, with agitator.

Test work has indicated that leach kinetics are enhanced with the inclusion of a pre-aeration step. The CIL circuit design will have a pre-aeration tank to facilitate oxidation/neutralization of any cyanide-consuming species as well as serving as the tank to enable the pH to be adjusted with lime to the required value of 10.5. The CIL feed slurry will exit the regrind circuit and enter the CIL feed distributor where the lime will be added.

The slurry will be gravity-fed into the pre-aeration tank from the CIL feed distributor. The pre-aeration tank will be equipped with a downcomer to ensure that the slurry enters the mixing area at the bottom of the tank. The tank will also have baffles and an agitator for the efficient mixing of the slurry and the reagents, and it will be equipped with air injection facilities to ensure that the required dissolved oxygen content will be reached and be maintained. The pre-aeration tank will have a retention time of about nine hours. Results from laboratory tests indicated that a conditioning/pre-aeration time of between eight and 12 hours is adequate to ensure oxygenation of the slurry for the cyanidation reaction to proceed.

The pre-aerated slurry will overflow the tank and will be discharged into the downcomer of the first CIL tank.

The pre-aeration tank can be by-passed if operational requirements dictate that this may be necessary.

### 17.3.10 CIL Circuit: Carbon Circuit

The gold and silver will be leached with cyanide and adsorbed onto activated carbon in this section of the processing circuit. Loaded carbon will be recovered periodically, nominally on a daily basis, for eluting/stripping to recover the gold and silver, and reactivated carbon will be added to replenish the loaded carbon in the circuit. The main items of equipment will be the following:

- CIL tanks, 6 x 10.0 m diameter by 10.6 m, each equipped with an agitator
- carbon transfer pumps
- NKM interstage screens
- loaded carbon screen
- carbon safety screen
- pumpboxes and pumps
- spillage sump pump.

The slurry will flow directly from from the pre-aeration tank into the first of six CIL tanks. Lime will be added if required to maintain pH control. Cyanide solution will be added to the first CIL tank, although provision will be made for cyanide addition to the downstream CIL tanks as well in order to maintain the concentration required for leaching. The cyanide concentration required in the CIL circuit will be about 2,000 mg/L as sodium cyanide, with about 200 mg/L sodium cyanide remaining in the CIL residue slurry.

Each CIL tank will be equipped with a downcomer, baffles, and an agitator. Each CIL tank will also be equipped with air injection facilities to maintain the dissolved oxygen concentration at an acceptable concentration of more than more than 5 ppm. The leaching of the gold and silver will proceed down the train of tanks with the overall cyanide leach circuit retention time being about 53 hours.

Each CIL tank will also be equipped with an interstage NKM-type screen and a recessed impeller slurry pump. The screens will retain the carbon in the respective CIL tank while permitting the pulp to flow through the screen to the next CIL tank in the circuit. The interstage screen aperture size will be 1,000 µm.

The carbon concentration in each tank will be 20 g/L. Presently, the requirement is to transfer an amount of about 3.5 t loaded carbon for elution from the head tank in the CIL circuit on a daily basis, generally within a four-hour period. However, the circuit will be designed to treat 5 t of loaded carbon with the equivalent slurry volume. Also, an equivalent amount of regenerated and/or fresh carbon will be added to the final tank in the CIL circuit every day. The recessed impeller pumps will transfer this regulated amount of slurry on a daily basis. The pulp transfer cycle will be set by the plant operations but could be done sequentially starting with the tail CIL tank or could be done by initially transferring slurry from all the even-numbered tanks (or odd-numbered tanks), followed by the odd-numbered tanks (or even-numbered tanks). Transfer durations will be conducted as required in order to maintain the carbon concentration at the nominal 20 g/L.

Loaded carbon will be pumped from the first CIL tank to the loaded carbon screen. The recessed impeller pump will be used to transfer the slurry containing the carbon. The loaded carbon screen will be a vibrating screen equipped with spray water nozzles to wash the adhering solids off the leached carbon. The loaded carbon will be transferred to the carbon circuit for acid treatment, elution and regeneration. The screen underflow will contain the slurry and wash water and will be returned to the first CIL tank.

The leached slurry will exit the CIL circuit via the interstage screen on the last CIL tank and will be discharged over the carbon safety screen. This vibrating carbon safety screen will also be equipped with spray water nozzles and will prevent any carbon particles, which may have by-passed the last interstage screen or been lost through a holed screen, from reporting to the residue. The carbon safety screen aperture size will be 600 µm. Any carbon which reports to the carbon safety screen will be collected in a bin and will be returned to the CIL circuit, or treated. The carbon safety screen undersize slurry will be collected in the carbon safety screen pumpbox and will be pumped to the detoxification circuit.

### *17.3.11 Cyanide Detoxification*

The cyanide content of the CIL circuit discharge will be reduced to acceptable concentration levels in the cyanide detoxification tank prior to discharge as a portion of the plant tailings.

The main items of equipment will be:

- cyanide detoxification tank with agitator
- air supply system
- reagent supply systems
- oxidation-reduction potential meter.

The CIL circuit tailings will be pumped to a detoxification tank to reduce the cyanide concentration to an acceptable environmental level prior to tailings disposal. The slurry will be pumped into the cyanide detoxification tank and reagents and air (oxygen) will be added to reduce the cyanide concentration from about 200 mg/L to less than 1 mg/L sodium cyanide. The reagents that will be added will include copper sulphate, and sodium metabisulphite (SMBS) which will generate the sulphur dioxide required for the cyanide detoxification reaction, as well as lime which will be added as required to maintain an alkaline pH for optimum cyanide detoxification. The cyanide detoxification tank will be equipped with air addition points as well as with an agitator to enable the air and the reagents to be thoroughly mixed with the tailings slurry.

The slurry will reside in the tank for a minimum of four-hours retention time, which is regarded as a conservative duration of time required for the cyanide concentration to be reduced to the acceptable environmental limits. An oxidation-reduction potential meter will be used to monitor the degree of oxidation of the cyanide. The overflow from the cyanide detoxification tank will be directed to the tailings dam feed pumpbox, and then pumped to the tailings dam standpipe.

From the tailings dam standpipe, the thickened and detoxified slurry will be pumped to the tailings storage facility for final deposition. This tailings product will be kept separate from the flotation tailings product which constitutes the bulk of the process plant tailings produced.

### 17.3.12 TAILINGS HANDLING

The flotation tailings and the CIL tailings will be piped separately from the processing plant to the tailings storage facility. The main items of equipment will be the following:

- tailings pumpboxes
- tailings pumps.

The tailings deposition and operation of the TSF will be the responsibility of the tailings management consultant.

The detoxified CIL tailings stream will be pumped directly to the tailings pond for deposition. This tailings stream will enter the tailings facility in such a way that deposition of the solids will always allow for the tailings to be submerged. Process water will be reclaimed from the dam via a reclaim water barge and pump system with the water reporting to the process water tank for distribution as required in the process plant.

Water for initial start-up and plant first-year operations will be provided from the tailings management facility starter dam and will be handled by the tailings management consultant.

### 17.3.13 CARBON CIRCUIT: CARBON HANDLING

The carbon handling circuit will include all the components necessary to move, store, add, and remove carbon in the carbon system. Carbon will be transferred between the various unit operations in the carbon plant by a screw type recessed impeller type pump (high clearance low degradation), and by pressurization of the elution column. Carbon transfer in the adsorption circuit will be by air operated valves and a high clearance pump. The valves will be operated locally or remotely from the PLC in the control room.

A carbon conditioning circuit will also be included in the circuit which will prepare the fresh carbon for use in the adsorption process and remove undersize carbon from the carbon circuit.

### 17.3.14 CARBON CIRCUIT: ACID WASHING

Loaded carbon will enter the acid wash tank from the loaded carbon screen. Under normal operating conditions, the loaded carbon will first be acid treated, then neutralized with caustic solution, or thoroughly rinsed, followed by the subsequent elution stage.

The acid wash circuit will have the following equipment.

- loaded carbon screen
- acid wash tank
- carbon transfer pump
- acid solution pumps
- acid wash pumpbox.

The acid wash tank will receive a batch of 3.5 t of loaded carbon for acid washing prior to eluting. A 3% strength hydrochloric acid solution will be re-circulated through the bed of carbon in the acid wash tank. This acid washing treatment will remove scale build-up and other inorganic contaminants which will inhibit gold adsorption onto the carbon. The duration of the acid wash will be about four hours. This will be followed by a neutralization wash and a water rinse to remove the remaining traces of hydrochloric acid.

The acid-washing step can be by-passed, if operational circumstances permit, with the loaded carbon being transferred directly to the elution column.

### 17.3.15 Carbon Circuit: Loaded Carbon Elution/Desorption/Stripping

The gold and silver adsorbed onto the carbon will be desorbed/eluted in the elution column. The elution circuit will have the following equipment.

- elution column
- carbon transfer pumps
- solution pumps
- hot water boiler with heat exchangers
- solution tanks
- dewatering screen
- solution samplers.

After the acid washing step, the loaded carbon will be pumped to the elution column. The elution solution will be heated using a combination of heat exchangers and a hot water heater.

After reaching the elution/stripping temperature, the solution will flow upward through the elution column. The elution column/strip vessel will be designed to treat a 5 t batch of loaded carbon. The gold-bearing solution will exit the elution vessel, and will flow through the two cool-down heat exchangers and then onwards by gravity to the electrowinning circuit.

The elution/stripping of gold and silver from the loaded carbon will be accomplished using the basic Zadra elution process. This process consists of continuously circulating a hot caustic/cyanide solution containing 1.0% sodium hydroxide and 0.1% sodium cyanide through the packed bed of carbon in the elution column. This solution will be heated to 140°C and the column will operate at a pressure of 375 kPa and up to a maximum of 525 kPa. A hot solution boiler will heat the solution to the required temperatures. The gold will be eluted from the carbon under these temperature and pressure conditions to form a pregnant solution.

Two cool-down heat exchangers will then cool the pregnant solution before the solution is transferred to the pregnant solution tank. The pregnant solution will then be pumped from the pregnant solution tank to the electrowinning cells to recover the gold and silver onto cathodes. The solution leaving the electrowinning cells, or barren solution, will be pumped to the barren solution tank. Sodium hydroxide and/or cyanide may be added to the barren solution to maintain the solution strength with respect to caustic and cyanide concentration. The barren solution will then be pumped through the heat exchangers to the elution column again until the elution cycle has been completed. The design solution flow rate for the elution process will be two bed volumes per hour. A typical elution cycle, including loading and unloading of the elution column, will take about 12 hours to complete. Two full elution cycles could be carried out in a 24-hour period, if necessary.

After the elution cycle is complete, the carbon will be transferred to the reactivation kiln feed bin via a dewatering screen.

### *17.3.16 Carbon Circuit: Electrowinning*

The gold and silver will be recovered by electrowinning from the pregnant solution. The electrowinning circuit will have the following equipment.

- electrowinning cells with anodes and cathodes
- sludge/cathode washing tank
- filter press
- filter feed pump.

The pregnant solution will be pumped from the pregnant solution tank to the electrowinning cells. The electrowinning circuit will consist of two 4.0 $m^3$ electrolytic cells. The gold and silver will be electroplated onto stainless steel wool cathodes. The average total daily metal recovery will be about 21 kg gold and silver. Barren electrolyte solution exiting the electrolytic cells will flow by gravity back to the barren/strip solution tank for the making up to strength with caustic and cyanide for the following elution/stripping cycle, namely to 1.0% sodium hydroxide and 0.1% sodium cyanide.

The flow rate of the pregnant solution through the electrowinning cells will be about 14.4 $m^3$/h, or two bed-volumes per hour. On completion of the electrowinning process, the electrowinning cells will be drained and the deposited metal will be washed off the cathodes. The cathodes can also be lifted out of the cells and placed inside the cathode wash tank for the thorough washing and removal of deposited metal, if necessary. The cell sludge and the metal washed off the cathodes will be washed into the cathode wash tank after every elution and electrowinning cycle. The sludge and metal will then be pumped to the electrowinning sludge filter press to remove the bulk of the solution, which will either be re-used in the electrowinning circuit, or recycled to the leach circuit. The filter press will be cleaned at the end of the filtration cycle. The metal and sludge will be placed into trays and dried in the drying oven, and then smelted. This frequency will be based on the operational campaign requirements.

### 17.3.17 Carbon Circuit: Carbon Reactivation

Carbon regeneration, or reactivation, will be conducted in a rotary kiln at a temperature of between 725 and 750°C. Eluted carbon will be regenerated/reactivated prior to its return to the CIL circuit. The reactivation section will have the following equipment:

- eluted carbon dewatering/sizing screen
- reactivation kiln
- kiln feed bin
- kiln screw feed conveyor
- carbon quench tank
- reactivated carbon sizing/dewatering screen
- reactivated carbon storage bin
- carbon transfer pumps.

The eluted carbon will be transferred hydraulically to the reactivation kiln carbon storage bin via a dewatering screen. This vibrating screen has a polyurethane deck with 1,000 µm slots and will be equipped with water spray nozzles. This will allow for the removal of any fine carbon particles which were generated during the carbon transfer and elution processes. The screen underflow will be collected in the fine carbon tank.

The kiln feed bin will be sized to store two batches of eluted carbon. At the design capacity of 3.5 t (i.e. 6.9 t of carbon), this will allow for storage in the event of a breakdown in the operation of the kiln. A by-pass facility will also allow eluted carbon to be transferred hydraulically to the carbon storage bin via the reactivated sizing screen thereby enabling the reactivation stage to be by-passed, if necessary. Both storage tanks will be equipped with drain screens to enable excess water to be drained from the carbon.

An inclined screw conveyor will feed the carbon to the rotary reactivation kiln while also allowing for some additional drainage of water from the carbon to take place.

Reactivation or regeneration of the eluted carbon includes the removal of organic adsorbates which have accumulated during the adsorption process. This process will restore the porous structure of the activated carbon. This reactivation process will be done using a rotary kiln. The reactivation kiln will be 1,200 mm in diameter and 7,000 mm long. The kiln will have an adjustable slope of between 0 and 4° with the pivot end at the feed side of the kiln. The nominal operating temperature of the kiln will be about 725°C with a maximum of 750°C. The tube of the reactivation kiln will be manufactured from 316L stainless steel, or suitable equivalent. The carbon entering the kiln will initially be wet and the kiln will first dry the carbon. The steam-carbon gasification reaction will subsequently oxidize the carbonized adsorbate residues as part of the regeneration process. The carbon throughput capacity of the kiln will be about 225 kg/h, with a retention time of 30 minutes, which, at the operating temperature of 725°C, will be sufficient for the proper reactivation of the carbon.

The reactivated carbon will exit the kiln and drop directly into the quench tank. The quench tank will be filled with water and the hot carbon will be rapidly quenched from the kiln temperature of about 725°C to about 75°C. The volume of the quench tank will be 9.2 $m^3$ with the dimensions 2,000 mm diameter and 2,300 mm high of which 300 mm will be freeboard for the inevitable splashing and agitation resulting from the dropping of the hot carbon into the water-filled tank.

Since the quench tank is only about 30% of the total volume required for the carbon from a complete elution of 3.5 t, the reactivated carbon will be transferred continuously by pumping to the reactivated carbon storage bin. The carbon will first be dewatered and sized over a reactivated carbon sizing/dewatering screen. This vibrating screen will have 1,000 µm slotted apertures. The reactivated carbon storage bin will have the capacity to store 5 t of regenerated carbon, equivalent to a volume of about 13.5 $m^3$. The dimensions of the tank will be 2,600 mm diameter and 3,200 mm in height, with a freeboard of 600 mm. The sizing/dewatering screen will be equipped with water spray bars to ensure that fine carbon particles (less than 1,000 µm) will be removed from the regenerated carbon before this regenerated carbon is returned to the CIL circuit. These fine carbon particles will be transferred to the flotation tailings, although provision could be made to recover the carbon fines using a fine carbon tank and small filter press.

### *17.3.18 Refinery*

The gold and silver will be smelted into bars. The refinery will have the following equipment:

- drying oven
- flux mixer
- induction smelting furnace
- bullion safe (or gold vault)
- dust and fume collection system.

The metal and sludge will initially be dried in the drying oven. This material will then be mixed with fluxes—typically a combination of borax, niter and possibly silica sand. Smelting will take place in a induction smelting furnace, equipped with hydraulic control of the tilt mechanism for pouring molten gold-silver melt product. A cascading mold system will be included. The bullion bars will then be cleaned to remove adhering slag, dried and weighed on a bullion balance, and then stored in the bullion safe until the bars are despatched to their final destination. The frequency of smelting will be as required, but will probably be once or twice every week.

A sample will be drilled from each bullion bar produced and submitted for assay in order to accurately determine the gold and silver content of each bar produced. The assays obtained will be correlated with the assays obtained from the facility buying the bullion for final reconciliation of the gold and silver produced, and the revenue earned from the bullion produced.

### 17.3.19 REAGENT HANDLING AND STORAGE

Various chemical reagents will be added to the process slurry streams to facilitate the recovery of sulphide minerals and gold during the flotation process and the subsequent leaching of gold during the CIL process, as well as the cyanide detoxification of slurry prior to final disposal. The preparation of the various reagents will require:

- a bulk handling system
- mix and holding tanks
- metering pumps
- a flocculant preparation facility
- a lime slaking and distribution facility
- eye-wash and safety showers
- applicable safety equipment.

Fresh water will be used in the making up or the dilution of the various reagents that will be supplied in powder/solids form, or which require dilution prior to the addition to the slurry. These solutions will be added to the addition points of the various circuits and streams using metering pumps.

To ensure spill containment, the reagent preparation and storage facility will be located within a containment area designed to accommodate 110% of the content of the largest tank. In addition, each reagent will be prepared in its own bunded area in order to limit spillage and facilitate its return to its respective mixing tank. The storage tanks will be equipped with level indicators and instrumentation to ensure that spills do not occur during normal operation. Appropriate ventilation, fire and safety protection, and Material Safety Data Sheet (MSDS) stations will be provided at the facility.

#### LIME

The process design includes the use of quick lime as the pH modifier for the various unit processes. Quick lime will be delivered in 40 t trucks. It will be added to the grinding circuit—specifically, the CIL circuit for protective alkalinity for the cyanide, as well as to the cyanide detoxification tank. The lime will be off-loaded into a storage silo. A screw conveyor will add the required amount of lime to the lime slurry mixing system. The lime slurry strength will be 20%. The lime will then be distributed to the addition points via a closed loop piping system.

#### POTASSIUM AMYL XANTHATE

PAX will be used in the flotation circuit as the collector reagent for the flotation of gold and sulphide minerals. The PAX will arrive at the plant in bulk bags and will be dumped into suitable sized hoppers for withdrawal of pre-determined quantities for mixing with water to the required solution strength of 20%. The reagent will be made up in a mixing tank, and then transferred to the holding tank, from where the solution will be pumped to the addition points in the circuit using metering pumps.

#### METHYL ISOBUTYL CARBINOL

The reagent MIBC will be used in the flotation circuit to provide the froth phase of the flotation process. MIBC will arrive in bulk containers and will be transferred to a holding tank prior to distribution within the plant. MIBC will be pumped directly to the flotation circuit as a 100% concentration strength solution using metering pumps.

#### FLOCCULANT

Flocculant will be used in the flotation concentrate thickener as an aid in the settling process. The flocculant will be prepared at the required concentration in a proprietary vendor-supplied flocculant preparation facility. Flocculant will be delivered in bulk bags. A screw conveyor will deliver the correct amount of dry flocculant powder to be mixed with water prior to delivery into the flocculant mix tank. The flocculant will be allowed to hydrate in the mix tank before being transferred to the holding tank where it will be made up to the required dosing strength. A metering pump will transfer the required amount of flocculant from the holding tank to the point of addition at the flotation concentrate thickener where final dilution will occur.

### SODIUM CYANIDE

The sodium cyanide will be delivered in bulk boxes (or drums) as small briquettes. Sodium hydroxide or caustic will initially be added to the cyanide mixing tank to ensure that the solution will be alkaline. The cyanide will be dissolved in water to the required concentration strength of 20% in a mix tank. The cyanide solution will then be transferred to the holding tank from where it will be distributed to the points of usage.

The cyanide preparation area will be isolated and only approved personnel will be allowed to enter the preparation area. This area, and the intensive cyanidation area, will be equipped with a hydrogen cyanide monitors to provide warning in the event that hydrogen cyanide gas is present.

### CARBOXYMETHYL CELLULOSE

The CMC will be added as a depressant of the carbonaceous material in the flotation circuit. CMC will be delivered in bulk bags, and will be prepared in the same manner as flocculant although the CMC solution strength will be 10%. A metering pump will deliver the CMC solution to the flotation circuit.

### HYDROCHLORIC ACID

Hydrochloric acid will be used for the dissolution of acid-soluble contaminants on the loaded carbon in the acid washing process. Typically this is calcium which has precipitated as calcium carbonate in the pores of the activated carbon. The hydrochloric acid will be delivered as concentrated acid. A sufficient amount of acid will be pumped from the acid containers directly to the water-containing acid wash pumpbox where the concentration will be adjusted to approximately 3% acid strength.

### SODIUM HYDROXIDE

Sodium hydroxide will be delivered in bulk bags. It will be mixed with water to make up batches of caustic solution at the required solution strength of 30%. The caustic solution will then be transferred to the caustic storage tank from where it will be pumped to the cyanide make-up circuit and the elution circuit, as required.

A secondary amount of sodium hydroxide will be available for make-up at the intensive cyanidation leach circuit.

### SODIUM METABISULPHITE

SMBS will be supplied as a solid material in bulk bags. It will be mixed with water to a 20% solution strength in the mixing tank. This solution will be pumped from the mixing tank to the holding tank. The solution will then be pumped to the cyanide detoxification tank at the required dosage rate.

#### COPPER SULPHATE

Copper sulphate will be supplied in the pentahydrate form as a solid crystalline material which will be supplied and shipped in bulk bags. It will be mixed with water to a 10% solution strength in the copper sulphate mixing tank. The copper sulphate solution will be pumped from the mixing tank to the cyanide detoxification tank at the required dosage rate.

#### ACTIVATED CARBON

Activated carbon will be delivered in bulk bags. The carbon will be introduced into the carbon conditioning tank where the slurry will be conditioned by removing the jagged edges of the carbon particles and the adhering carbon dust. The slurry will be pumped over the sizing/dewatering screen with the coarse carbon particles added to the CIL circuit, and the carbon fines discharged to the tailings stream. The carbon will be added to the carbon conditioning tank as required and pumped to the carbon sizing/dewatering screen for transfer to the CIL circuit by pumping via a holding tank, as required.

### 17.3.20 ASSAY AND METALLURGICAL LABORATORY

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

- fire assay equipment
- atomic absorption spectrophotometer (AAS)
- x-ray fluorescence spectrometer (XRF)
- Leco furnace.

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

### 17.3.21 WATER SUPPLY

Three separate water supply systems for fresh water, treated water and process water will be provided to support the operation.

#### Fresh Water Supply System

Fresh and potable water will be supplied to a fresh/fire water storage tank from a suitable supply source yet to be identified, but probably from wells. Fresh water will be used primarily for the following:

- fire water for emergency use
- cooling water for mill motors and mill lubrication systems
- gland service for the slurry pumps
- reagent make-up
- potable water supply.

The fresh/fire water tank will be equipped with a standpipe which will ensure that the tank is always holding at least 40 $m^3$ of water, equivalent to a two-hour supply of fire water.

The potable water from the fresh water source will be treated and stored in the potable water storage tank prior to delivery to various service points.

#### Process Water Supply System

Some process water generated in the flotation circuit as concentrate thickener overflow solution will be re-used in the process circuit via the process water tank. Reclaimed water will also be pumped from the tailings impoundment area to the process water tank for distribution to the points of usage.

#### Treated Water Supply System

The gravity concentrators require clean, fresh water for fluidisation purposes. However, the volume requirement for suitable fresh water is relatively significant. This could result in environmental permitting issues since the resulting excess process water will require costly treatment prior to its return as discharge to the environment. For this reason, a separate water circuit using treated process water for the gravity concentration circuit will be included in the design of the overall water system.

### 17.3.22 Air Supply

Air will be required for process use and instrumentation purposes. Separate air service systems will supply air to the following areas:

- low-pressure air for the flotation cells and CIL tanks will be provided by air blowers.

- instrument air will come from the plant air compressors and will be dried and stored in a dedicated air receiver.
- high-pressure air for plant air distribution and for the instruments will be provided by air compressors.

Three air compressors will supply plant and instrument air to the process plant. The air from the compressors will be fed to an air receiver and will pass through an air filter to remove remnant grease or oil. The air will be used for general purpose plant compressed air requirements.

An off-take from the discharge line after the plant air filter will be passed through the instrument air filters and then through a dryer unit before being fed to the instrument air receiver. This will provide the air volume and pressure required by the plant instruments.

Three centrifugal air blowers will provide the dedicated low-pressure air required by the flotation circuit, as well as providing the air required for the tanks in the pre-aeration, CIL and cyanide detoxification circuits.

### *17.3.23 SAMPLE ANALYSIS*

The plant will not rely on on-stream or in-stream analyzer for process control. Specific samples that will be taken for metallurgical accounting purposes will be the flotation feed to the circuit, the final flotation tailings, and the CIL circuit feed and tailings. These samples will be collected on a per shift basis, and assayed in the assay laboratory. On-stream particle size monitors will determine the $P_{80}$ particle size of the primary cyclone overflow and the regrind circuit products.

# 18.0 PROJECT INFRASTRUCTURE

## 18.1 OVERVIEW

The Project will require the construction of a number of infrastructure items, including:

- a primary crushing building
- a coarse crush stockpile
- a pebble crushing building
- a mill building
- an administration and mine dry building
- a maintenance and truck shop building
- an assay laboratory
- a cold storage warehouse.

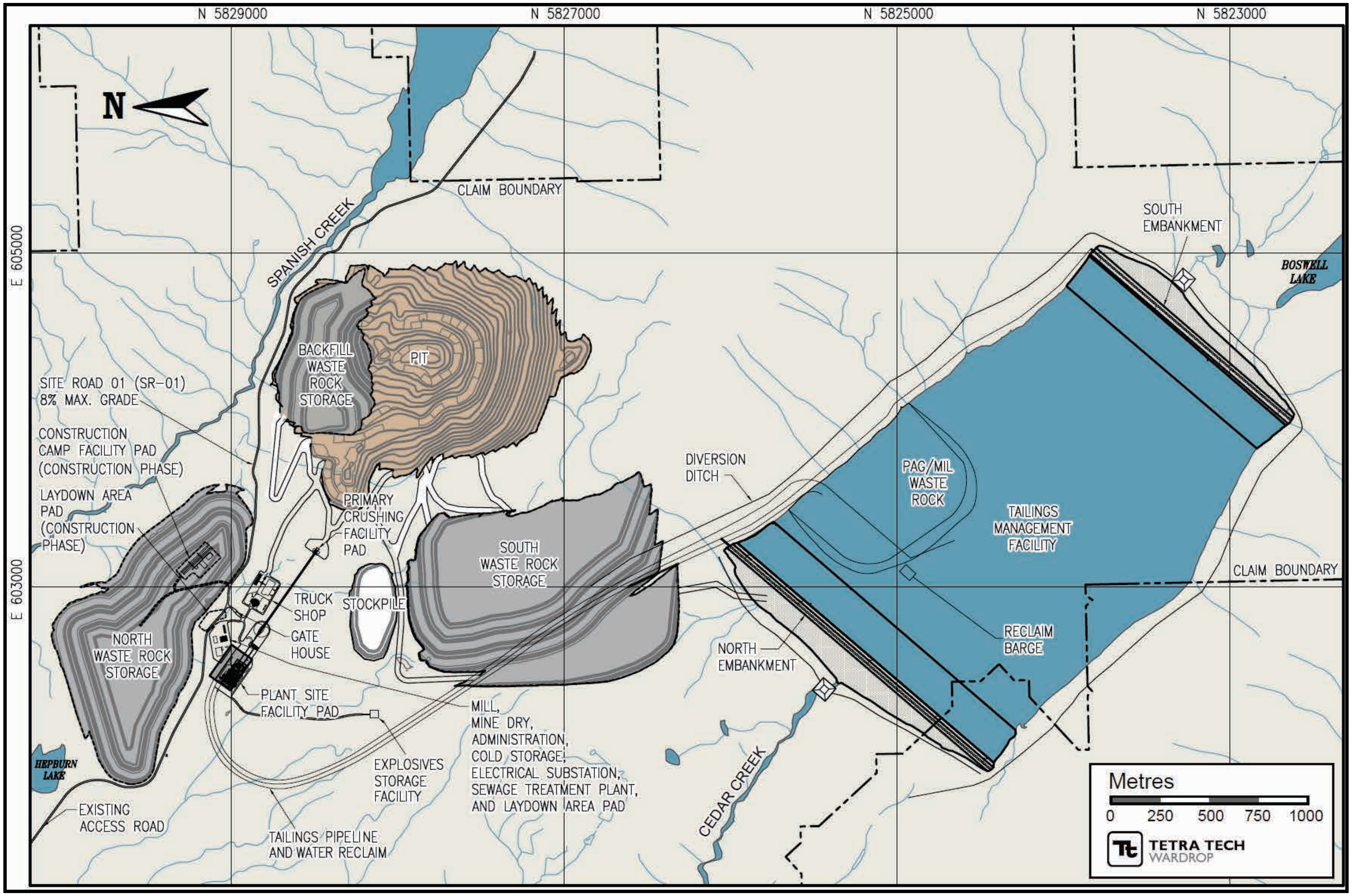
All buildings and facilities will be constructed with appropriate HVAC and fire protection systems, water and plumbing systems, and fire protection and dust control systems. A series of mine haul roads will be constructed from the open pit to the primary crusher, and as well as site roads to and from the truck shop, tailings storage areas, and WRSAs.

## 18.2 SITE LAYOUT

The recommended locations of various project facilities take advantage of the local topography and the TSF. Furthermore, the final site layout has been configured to ensure efficient and convenient operation of the mine haul fleet for minimum haulage to the primary crusher and the tailings embankments. The coarse crush material from the primary crusher will be conveyed by the overland conveyor to the coarse crush storage facility and from there the coarse material will be conveyed to the SAG mill for processing.

The location of the main project facilities is shown in Figure 18.1.

**Figure 18.1 General Arrangement Layout**



## 18.3 ROADS

### 18.3.1 ACCESS ROAD

From Williams Lake, the Property can be reached via a paved secondary road that leaves Highway 97 at 150 Mile House, approximately 16 km south of Williams Lake, and continues for 87 km to the community of Likely. From Likely, the Property can be accessed from the Spanish Mountain 1300 FSR. This road currently travels through the proposed mine site; it will require rerouting in order to accommodate the location of the open pit and waste dumps. Access to this FSR route through the site will be maintained throughout the LOM.

### 18.3.2 SITE ROADS

The mine site is spread over a distance of approximately 6 km. A network of site roads connecting the pit and the various on-site facilities will be constructed throughout the site.

Major site roads will include mine haul roads from the open pit to the primary crusher, and roads to and from the truck shop, tailings storage areas, and WRSAs. These major roads will generally be gravel-surfaced and 30 m wide. The road sub-base and base requirements will be governed by the quality of the subgrade; overall road thickness is expected to be approximately 1 m. The maximum road grade will be 10%. Safety berms will be provided wherever needed.

Other site service roads will interconnect the following facilities and areas:

- open pit
- process plant
- coarse crushed stockpile
- main substation
- primary crusher/stockpile
- maintenance facilities
- construction camp
- explosives storage
- overland conveyor right of way.

These site service roads will also connect to the mine fleet haul roads, providing explosive supply trucks direct access to the pit. These roads will generally be 6 to 8 m wide and gravel-surfaced. The roads will be constructed with a maximum grade of 10%.

## 18.4 BUILDINGS

### 18.4.1 PRIMARY CRUSHING BUILDING

The multi-level primary crushing building will be constructed of concrete, and will house the gyratory (primary) crusher, the primary apron feeder and the primary discharge conveyor.

The structure will be earth-retaining on three sides. ROM coarse crush will be discharged into the dump pocket at the top level. Interior steel platforms will be provided to support equipment for ongoing operation and maintenance. The control room adjacent to the dump pocket will be a modular pre-fabricated unit.

### 18.4.2 COARSE CRUSH STOCKPILE

The coarse crush stockpile will be fed by the primary crusher discharge conveyor. The concrete reclaim tunnel with four belt feeders, will discharge onto a SAG mill feed conveyor.

### 18.4.3 PEBBLE CRUSHING BUILDING

The pebble crushing building will be an engineered post and beam steel structure with insulated steel roof and walls. The building foundation will comprise concrete spread footings, grade walls along the building perimeters, and slab-on-grade. The building will be serviced with a 25 t overhead crane.

The two pebble crushers will send the fine crush to the SAG mill via a SAG mill feed conveyor.

### 18.4.4 MILL BUILDING

The mill building will be an engineered post-and-beam steel structure with insulated steel roof and walls. The building foundation will comprise concrete spread footings, grade walls along the building perimeters, and slab-on-grade floor. The floor surfaces will have localized areas sloped toward sumps for cleanup activities.

Interior steel platforms on multiple levels will be provided for ongoing operation and mill facility equipment maintenance. The building will house milling, media separation, refining, flotation, reagents, electrowinning, and some offices for mill staff. Several tanks will be placed externally but adjacent to the mill building, including the CIL feed thickener, cyanide detoxification, CIL circuit tanks, process water tank, and fresh/fire water tank. The building will be serviced by three cranes of various capacities that match specific functions.

### 18.4.5 Conveying

Conveyors will be vendor-supplied systems that will include all structural support frames, trusses, bents and take-up structures. Overland conveyors will be supported on concrete pre-cast strip panels/sleepers spaced at regular intervals. Elevated conveyor systems will be supported on vendor-supplied steel trusses spanning between steel bents on concrete spread footings.

### 18.4.6 Administration and Mine Dry

The administration and mine dry complex will be a prefabricated modular structure located close to the mill building. This facility will house a mine dry and 228 lockers to service 114 mine personnel per shift, a mill dry and 150 lockers to service 75 mill personnel per shift, and office areas for staff and supervisors. A mud room will also be attached.

### 18.4.7 Maintenance and Truck Shop

The truck shop building will be a pre-engineered steel building with insulated roof and walls. The building will be supported on concrete spread footings with concrete grade walls along its perimeters. Sumps and trenches will be constructed to collect waste water in the maintenance bays. Floor hardener will be applied to concrete surface for high traffic areas.

The building will house a wash bay complete with pressure washer, three repair bays, warehouse area, warehouse/parts storage, welding area, machine shop, emergency vehicle parking, first aid room, electrical room, mechanical room, compressor room and a lube storage room. The warehouse and repair bays will be serviced by two 10 t overhead cranes.

### 18.4.8 Assay Laboratory

The assay laboratory will be a pre-fabricated modular structure located close to the mill building. This facility will house all necessary laboratory equipment for metallurgical grade testing and control. The lab will be equipped with all appropriate HVAC and chemical disposal equipment as needed. The facility floor will be reinforced as needed to accommodate specialized equipment.

### 18.4.9 Cold Storage Warehouse

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

## 18.5 Building Services

### 18.5.1 HVAC and Fire Protection

All process areas will be heated to a minimum temperature of 5°C on a typical winter day, by providing multiple propane-fired heating units along the perimeter walls and above all doorways.

Large process buildings will be ventilated year-round to prevent a buildup of contaminants and humidity. The ventilation rate will rise during summer months. During winter months, air will be exhausted from the upper portion of the building using multiple exhaust fans, and propane-fired makeup air units will filter the outdoor air and temper it before drawing it into the building through ductwork installed within 4 m of the ground floor.

All staff-occupied areas, such as offices, washrooms, and change rooms, will be heated to a minimum temperature of 20°C on a typical winter day, by supplying filtered and tempered outdoor air mixed with return air. The air will be distributed through ductwork into the individual rooms.

Air conditioning will be limited to control rooms, laboratories, and electrical rooms in which heat generated from electrical equipment is expected to be excessive. Electrical rooms in which heat is not expected to be significant will be cooled with filtered outdoor air. Small rooms, electrical rooms, and remote buildings will be heated using electric heat.

Washrooms, change rooms, print rooms, janitor's rooms, showers and lunch rooms will be mechanically exhausted to atmosphere. Make-up air will be transferred from adjacent areas, or be supplied with filtered tempered outdoor air.

### 18.5.2 Plumbing

All plumbing fixtures will be installed with hard-pipe components, and will gravity-feed to a sanitary drainage system. Sinks and showers will also be installed with hard-piped components to supply both potable hot and potable cold water. All plumbing fixtures connected to the sanitary system will be vented.

Water will be heated in hot water storage tanks by propane or electricity, and installed close to usage points. All cold water piping will be insulated to prevent condensation, and all hot water piping will be insulated for heat conservation.

Oil separators will be installed in truck shops and truck washes. Grease interceptors will be installed in construction camp kitchens.

### 18.5.3 FIRE PROTECTION

A fire water tank capable of sustaining enough water for two hours of firefighting at a flow rate equal to the largest sprinkler flow, plus flow from an inside hose and an outside hydrant. The fire water system will be pressurized by a fire water pump package comprising a jockey pump, a main electric pump, and a standby diesel-fired pump.

Yard hydrants will be positioned around the site so that all outside walls of buildings and fuel tanks can be reached by a 30 m hose and a 15 m hose stream.

Sprinkler systems will be provided in lube rooms, air compressor rooms, blower rooms, truck shops, warehouses, laboratories, the MIBC room, the elevated mill offices, the mining equipment storage building and the administration building.

### 18.5.4 DUST CONTROL

Dust control systems, comprising dry baghouses, will be installed at the primary crusher apron feeder, the coarse crush reclaim feeders, and the pebble crusher area.

The dust will be pneumatically conveyed from each exhaust hood to the dust collector through steel ducting, which will include test ports, dampers and cleanouts. Collected dust fines will be returned to the process stream.

Dust will also be controlled at the truck dump, using a series of water spray nozzles, whenever the outdoor temperature is above freezing.

## 18.6 WATER SUPPLY AND DISTRIBUTION

The construction camp will require approximately 320 L/person/d (or 80 $m^3$/d) of potable water to supply a peak of about 250 direct workers, staff and indirect operations personnel during the construction phase of the Project. A potable water package treatment unit, including a bottling facility for drinking water, will be located close to the administration building. The potable water will undergo chlorination and ultraviolet light treatment.

Fresh water will be pumped from local groundwater wells to fresh and firewater storage tanks, located near each major facility. To ensure adequate firewater levels, two individual reservoirs inside the tank will be separated by an internal standpipe. The tank will be equipped with a level detection control system.

The fire/fresh water tank for the process building will be located outside and will not be insulated. The water pressure will be maintained by booster pumps, jockey pumps and diesel back-up pumps. A plant site alarm will signal a low system pressure condition. The hydrant system comprise a wall-mounted with the pipe ring installed inside the process building.

The fire/fresh water tank for the truck shop/administration/construction camp area will be located inside the truck shop building, to prevent freezing during winter months. The emulsion plant and the primary crusher area will have a dedicated storage tank, well, and pumping system for fire water and fresh water, to prevent long buried lines from a central system.

Emergency showers and eyewash stations will be installed throughout the process building.

## 18.7 WASTE DISPOSAL

### 18.7.1 SEWAGE DISPOSAL

During the construction phase, 120 L/person/day (or 30 $m^3$/d) of sewage will be collected, approximately 40 L/person/d (or 10 $m^3$/d) of which will be treated. The proposed treatment plant will be a membrane bioreactor system and chemical phosphorus removal to meet 1 mg/L total phosphorus.

### 18.7.2 DOMESTIC WASTE DISPOSAL

Construction, industrial waste, and scrap metal will be collected in bins and recycled by a qualified local contractor. Domestic waste from the construction camp and operating areas will be transported to the local Likely, BC landfill.

Several forms of domestic and industrial solid waste will be generated over the life cycle of the mine. All avenues of re-use, reduction and recycling of materials will be examined and implemented prior to disposal of any waste.

## 18.8 FUEL STORAGE

The diesel fuel storage comprising two 500,000 L tanks will be housed in a high-density polyethylene (HDPE)-lined, dyked tank farm, located near the truckshop. Fuel dispensing facilities, including light vehicle as well as fast-fill facilities for mining equipment, will be included.

## 18.9 ON-SITE EXPLOSIVES STORAGE

### 18.9.1 EMULSION PLANT

The emulsion plant will be a 50 m by 50 m compound located south of the open pit, approximately 1,000 m from the nearest road or mine building. The compound will include:

- an emulsion plant
- emulsion storage.

### 18.9.2 Detonator and Explosive Storage Magazine

Two 9 $m^2$ Type 4 magazines will be fabricated from a 6 mm metal plate. The magazine walls will comprise at least 7.6 cm of bullet-resistant material; the roof will be a 4.7 mm (or thicker) metal plate. The magazines will be mounted on large metal I-beam skids, providing a minimum ground clearance of 10 cm or more for structural rigidity and portability. The magazines will be fabricated in accordance with current storage standards for industrial explosives.

## 18.10 Power Supply to Plant Site

The Project requires 60 MW of peak load for 40,000 t/d operation demand. An existing BC Hydro 69 kV transmission line 60L300 feeding the Mount Polley Mine cannot accept any additional power load. Therefore, a new transmission line interconnecting the SMG site to BC Hydro's power system is required to meet power requirement in operation.

Stantec performed an assessment for an interconnection transmission line to BC Hydro's power system for the Project. This assessment did not reflect BC Hydro's system impact study (SIS), which is currently underway. The final result of BC Hydro's SIS will determine the most suitable point of interconnection to BC Hydro's grid, and the estimated costs associated with system reinforcement.

### 18.10.1 Power Supply

Stantec evaluated six power supply options, including preliminary design basis, cost estimate, bill of materials and development schedule. Stantec's complete power supply evaluation is provided in Appendix H.

In April 2012, BC Hydro released the preliminary results of its SIS, which indicated that 69 kV supply is not technically feasible.

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

## 18.11 TAILINGS STORAGE FACILITY

### 18.11.1 *DESIGN BASIS AND OPERATING CRITERIA*

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

The metallurgical process involves a gravity circuit followed by a rougher flotation circuit to produce rougher tailings. Approximately 3% of the process feed is reground and subjected to carbon-in-leach and cyanide detoxification circuits before being combined with a prefloat concentrate comprising about 5% of the feed and cleaner tailings comprising about 2% of the feed to produce the overall cleaner tailings stream, which is assumed to be PAG and ML if allowed to oxidize. The tailings streams will be transported from the plant site to the TSF in separate pipelines at an average solids content of 38% by weight for the rougher tailings and 43% by weight for the cleaner tailings. Each tailings stream will be deposited independently; the rougher tailings will be discharged along the TSF embankments to create tailings beaches and the cleaner tailings will be discharged subaqueously in the supernatant pond and progressively encapsulated by the rougher tailings.

The TSF capacity at all stages of the mine life includes the supernatant pond volume and allowances for wave run-up (0.5 m), post-seismic settlement (0.25 m), sloping beaches (3 m) and containment of the inflow design flood (4.8 $Mm^3$).

### 18.11.2 *WASTE MANAGEMENT FACILITY EMBANKMENTS*

The TSF will comprise a north embankment and a south embankment. The embankments will be zoned earthfill/rockfill structures, with a low-permeability core for seepage management. The embankments will also include filter and transition zones to ensure proper filter relationships between adjacent zones, and to convey drainage within the embankment. A downstream shell zone comprises the majority of the embankment material.

The starter TSF—which will be constructed during the pre-production phase—has been sized to store the estimated volume of tailings and PAG/ML waste rock produced during the first two years of operation, plus the supernatant pond volume, and associated freeboard allowances. The TSF embankments will be constructed in stages; each stage will provide the required capacity for the period until the next stage of construction is completed. The final capacity of the TSF will be approximately 200 Mt of tailings, 35 Mt of PAG/ML waste rock, plus the supernatant pond volume and freeboard allowances.

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

### 18.11.3 Construction Materials

The TSF embankments will be constructed using suitable waste rock and overburden (low permeability glacial till) from the open pit. Additional overburden will be sourced from a borrow west of the open pit, as required.

### 18.11.4 Tailings Distribution Systems

The rougher tailings will be discharged into the TSF from a series of large diameter valved off-takes located along the embankments. Selective tailings deposition will be used to keep the tailings pond away from the embankments to reduce seepage losses from the TSF and encapsulate the cleaner tailings.

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

### 18.11.5 Reclaim System

Reclaim water for use in the mill processes will be pumped via a 18" diameter HDPE pipe from a floating barge on the TSF to a process water tank located outside of the mill building. The tank will store a 24-hour supply of mill process water. The barge will be positioned at the north end of the TSF to minimize pumping distance to the process water tank.

### 18.11.6 Water Management

The TSF supernatant pond will serve as a primary site water management component, providing a buffering for process water, direct precipitation and runoff.

Surface diversion ditches will capture and divert non-contact water around the TSF for release to the environment. Runoff from catchments directly upstream of the TSF will be diverted to Cedar Creek, while runoff from catchments upstream of the south embankment will be diverted to Boswell Lake, where it will be directed through an overflow channel to Winkley Creek and, eventually, to Quesnel Lake.

Seepage collection ponds and pumping systems are included downstream of each of the embankments to collect runoff and seepage from the embankments. Water from the seepage collection ponds will be pumped back to the TSF.

## 18.12 Waste Rock Management

### 18.12.1 Waste Rock Production

MMTS developed a 40,000 t/d production schedule which defined the amount of mineralized material, waste rock, overburden and undefined material produced annually over the mine life. MMTS also identified and categorized the waste rock, based on its geochemical characterization (see Section 16.5.1).

### 18.12.2 Waste Disposal Strategy

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

## 18.13 Construction Camp Accommodation

On-site camp accommodations for construction personnel will be provided during the construction phase. The current design includes:

- 250 single-occupancy rooms
- dormitory building
- kitchen
- recreation complex
- administration complex
- sewage treatment facility
- potable water facilities.

After construction is completed, the temporary camp will be dismantled.

## 18.14 Communications

The telecommunications design for the Project will incorporate proven, reliable, and state-of-the-art systems to ensure that personnel at the mine site will have adequate data, voice, and other communications channels available.

The telecommunications system will be supplied as a design-build package. The base system will be installed during the construction period, and then expanded to encompass the operating plant. The design will include:

- a Voice over Internet Protocol (VoIP) telephone system
- satellite communications for voice and data
- ethernet cabling for site infrastructure
- wireless Internet access
- two-way radio communications at site

- satellite TV.

A main telecommunications central equipment office will consist of a pre-manufactured trailer in which the main communications contractor will install and test all the main sub-systems for the facilities, prior to shipment. The trailer will form the first block in a system that must support the construction needs of the Project first, and the operating needs of the Project following construction.

Spare parts for critical and main components will be provided to ensure maximum reliability, and minimum down time. A variety of communications media (copper and wireless during the construction phase and fibre optic during the operating phase) will be incorporated in the overall design.

The requirements for communications, particularly satellite bandwidth, are a function of the voice and data requirements of the active participants in the Project. The expectation is that the need for satellite bandwidth will build to a peak during the plant construction phase, and then taper off slightly as the initial construction crew yields to plant operations.

Technologies and services to be provided include:

- Construction Phase:
  - local VoIP wireless network
  - satellite link for voice, data and video services
  - personal computer Local/Wide Area Network (PC LAN/WAN)
  - trunked mobile radio system
  - Internet service
  - private telephone system for voice and fax
  - video conferencing to minimize travel during design and construction
  - ground-go-air communications system (VHF radio)
  - cable television (CATV) on independent satellite system
- Operations Phase (includes selected services from the Construction Phase):
  - process monitoring and control for efficient operation and maintenance
  - fibre optic cabling for plant wide communications
  - security access control.

# 19.0 MARKET STUDIES AND CONTRACTS

The Project will yield gold doré as its final product, which is expected to be sold on the spot market through marketing experts retained by SMG. Gold can be readily sold on numerous markets throughout the world; its market price at any particular time is easily and reliably ascertained. The large number of available gold purchasers, both domestically and internationally, allow for gold production to be sold on a regular and predictable basis, and on a competitive basis with respect to the spot price.

SMG expects that terms contained within any potential sales contract would be typical of, and consistent with, standard industry practices.

# 20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

## 20.1 ENVIRONMENTAL STUDIES

Environmental studies—including studies on surface and groundwater quality and quantity, geochemistry, climatology, fish and fish habitat, wildlife, and vegetation—have been ongoing at the Project site since 2007.

Water quality monitoring sites have been established throughout the Project area to characterize existing water quality conditions. Water quality samples from within the claim boundary have consistently shown concentrations of total and dissolved metals that exceed limits set by the Canadian Council of Ministers of the Environment (CCME) and the BC Water Quality Guidelines (BCWQG) for the protection of aquatic life. The level of these concentrations is likely caused by the natural mineralogy of the claim area and historic placer mining activities.

Site-specific fish and fish habitat assessments 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 near the mouth of Cedar Creek; juvenile Chinook were captured and adult Coho salmon were detected near the mouth of Spanish Creek.

The Wells Grey 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 of the Project. The range of the Quesnel Lake North population of grizzly bear covers the Project area. Other flora and fauna species in the Project area are typical for the region.

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

## 20.2 WASTE AND TAILINGS DISPOSAL, SITE MONITORING, AND WATER MANAGEMENT

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

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

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 groundwater. Containment strategies for the waste material will be implemented to minimize air and water exposure of the reactive waste material. Drainage from waste rock storage areas and mine workings will be monitored for the life of the Project.

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

## 20.3 PERMITTING

A typical EA is generally completed in three to four years. For this Project, the EA process began on July 8, 2011, with the submission of a project description to the BC EAO and the federal CEAA. Detailed environmental and socio-economic baseline studies were then initiated; these typically require a two-year period to complete. These studies, along with any completed feasibility studies, will form the basis of an impact assessment, which will be submitted as part of the EA and reviewed by regulators, First Nations, and the public.

When this review is complete, a provincial EA certificate and federal Notice of Decision will be issued. SMG and consultants will then work with provincial and federal regulators to advance the required permits and authorizations. The required principal provincial permits are expected to be a *Mines Act* permit and an *Environmental Management Act* permit. Federally, the principal permits are expected to be an *Explosives Act* permit, authorization under the *Navigable Waters Protection Act*, and authorization under the *Fisheries Act*.

## 20.4 Social or Community Requirements

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

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

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

Community and First Nations consultation has been initiated by SMG and will continue throughout the development of the Project.

## 20.5 Mine Closure

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

# 21.0 CAPITAL AND OPERATING COSTS

## 21.1 SUMMARY

Tetra Tech developed a capital cost estimate (CAPEX) and operating cost estimate (OPEX) for the Project, based on the findings of this study. The CAPEX and OPEX are calculated in Canadian dollars.

The total estimated pre-production capital cost for the design, construction, and installation and commissioning for all facilities and equipment is CAD$763.1 million.

This estimate has been prepared in accordance with the Class 4 Prefeasiblity Cost Estimate standards of the AACE. The accuracy of the estimate is ±35%, unless otherwise noted.

The operating cost is estimated to be CAD$10.78/t, with an accuracy range of ±35%.

This study has been prepared with a base date of Q4 2012 with no provision for escalation.

Further details regarding the CAPEX can be found in Appendix I.

## 21.2 CAPITAL COST ESTIMATE

The CAPEX consists of four main elements:

- direct costs
- indirect costs
- Owner's costs
- contingency.

### 21.2.1 SUMMARY OF CAPITAL COSTS

A summary breakdown of the capital cost is provided in Table 21.1.

**Table 21.1 Capital Cost Summary**

| Description | Capital Cost CAD$ million |
|---|---|
| **Direct Costs** | |
| Overall Site | 20.1 |
| Open Pit Mining | 128.9 |
| Mineralized Material Handling | 54.8 |
| Process | 169.7 |
| Tailings and Water Management | 70.4 |
| Environmental | 12.0 |
| On-site Infrastructure | 57.0 |
| Off-site Infrastrucutre | 16.3 |
| **Subtotal** | **529.3** |
| Indirect Costs | 130.2 |
| Owner's Costs | 16.7 |
| Contingencies | 86.9 |
| **Total** | **763.1** |

Note: Numbers are rounded.

### 21.2.2 ESTIMATING METHODOLOGY

The following steps were followed to create this estimate:

- define work breakdown structure (WBS), code of accounts (COA), and estimate sequential codes (Seq.)
- research and calculate labour rates
- research and calculate productivity factors based on site and project conditions
- schedule work on a time/logic basis based on discussions between project teams
- obtain pricing for select equipment and materials from vendors and suppliers for all work under the scope of Tetra Tech
- estimate installation man-hours, based on inputs from suppliers and Tetra Tech experience
- estimate the cost of commissioning and capital spares
- establish requirements for freight, customs clearance, duty and taxes
- estimate costs for detail design, procurement and construction management required to perform these services
- establish foreign currency costs and exchange rates, as applicable
- establish an appropriate estimate base date.

### 21.2.3 Responsibility

Tetra Tech was responsible for the assembly of the overall estimate; the components of the estimate were developed with input from engineers, procurement specialists and cost estimators from the following companies:

- Tetra Tech: process, layout and general arrangement, plant infrastructure, dust control, building services such as HVAC (heat recovery) and fire protection, instrumentation and controls, piping, process plant electrical distribution, and mechanical equipment
- Knight Piésold: tailings and water management, and environmental
- MMTS: open pit mining
- Stantec: off-site infrastructure and external power supply (BC Hydro)
- SMG: Owner's costs, etc.

### 21.2.4 Direct Costs

#### Quantity Development

All quantities were derived from general arrangement drawings, process design criteria, process flow diagrams, and equipment lists.

All engineering measurements are exact as designed (i.e. no additional design allowances). No contingency has been built into the estimate for quanities.

Engineering, procurement and construction management (EPCM) and all contingencies have estimated separately by all disciplines and/or outside consultants, and included in the indirects portion of the estimate.

#### Pricing

Budget quotations were obtained for selected (major) equipment. Equipment vendors were asked to provide prices, delivery lead times, freight costs to a designated marshalling yard, and an allowance for spares where possible. The quotations provided were budgetary and non-binding.

Other items in the CAPEX are based on the following elements:

- Tetra Tech's in-house database for selected equipment costing has been based on in-house data or quotes from recent similar projects.
- Preliminary material take-offs were completed by each Tetra Tech discipline, and each consultant.

- All equipment and material costs are included as Free Carrier at the manufacturer's plant, per Incoterms 2010 definitions. Other costs such as spares, taxes, duties, freight and packaging are included in the indirect section of the estimate.

### 21.2.5 INDIRECT COSTS

The Project indirect costs include:

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

### 21.2.6 OWNER'S COSTS

The Owner's costs are estimated to be CAD$16.7 million. These costs include:

- home office staffing
- home office travel
- home office general expenses
- home office miscellaneous expenses:
  - project legal costs
  - product marketing
  - land taxes
  - reclamation bonds
  - project funding or financing costs
  - performance bond premiums
  - environmental baseline monitoring (stream gauging, water quality, air quality, weather station)
  - environmental permitting
  - development or building permits

  - relocation costs and assignment costs (moving, medical, visas, etc.)
  - project photographs
  - licenses
  - sales taxes
  - import duties and tariffs
  - builders risk insurance
  - general liability insurance
  - political risk insurance
  - marine insurance (import of overseas material)
  - miscellaneous allowances for deductible claims
  - geotechnical work and drilling programs
  - metallurgical test work programs
  - business permits and licenses
  - commissions and royalties
  - miscellaneous outside consultants
  - right-of-way and land purchase costs
  - sunk costs or acquisition costs
  - partnership or joint venture costs
  - goodwill and local infrastructure contributions
  - removal of hazardous material;
- field staffing
- field travel
- field general expenses
- training program development
- miscellaneous expenses:
  - mine modelling/planning
  - computer hardware/software
  - geological drilling
  - laboratory costs
  - housing and Owner's facilities costs
- Owner's contingency:
  - Owner's risk assessment
  - post start-up modifications.

### 21.2.7 CONTINGENCIES

The overall contingency for the Project is CAD$86.9 million. This contingency amount excludes scope changes, force majeure, escalation and currency fluctuations.

### 21.2.8 MINING CAPITAL COST ESTIMATE

An initial CAPEX was completed for the following mining components:

- Pre-production cost is estimated for 9.8 Mt of waste and 1.9 Mt of mineralized material mined in pre-production. The costs include hauling 7.2 Mt of suitable waste rock to the tailings embankment for construction. Mine development will be undertaken by a contractor and will consist primarily of haul road construction. It is anticipated that the Owner's mine equipment fleet will be available for all mining activities thereafter.
- Initial mine equipment includes the total fleet requirement to meet the total material production in Year 1. The equipment pricing is based on new units delivered to the mine, with all transportation and erection costs included. Most unit prices are based on recent vendor budgetary quotations. Others are sourced from the MMTS equipment database. Used equipment, if available, will reduce these equipment capital costs, and have not been considered for this study.
- Pit dewatering and depressurization costs are estimated based on the plan provided by BGC. It includes drilling vertical wells and horizontal holes, pump installations and maintenance. Costs for the pumps, pipes and power for operating the pumps are included elsewhere.
- Mine fleet will consist of diesel powered equipment and no electric power will be required in the pit. Power to operate pumps and depressurization wells will be from diesel generators.
- Site-preparation cost is an allowance for clearing and grubbing, drainage ditches, topsoil removal and acquiring granular materials for road surfacing.
- Road construction costs are estimated for approximately 11 km of haulroads to be undertaken by a contractor
- Salaries and costs for mine and engineering staff, and consultants during the pre-production period are included in the Owner's costs.
- The cost for the truck dispatch system is included with this estimate. Other capital cost allowances for computer supplies, surveying equipment, and communications facilities are included in the Owner's costs.
- Three percent of the mobile equipment fleet capital is included for spare parts such as truck tires, loading buckets, shovel teeth, drill bits, etc. Due to the proximity of the mine to other operating mines and service centres, it is anticipated that this amount carried at the mine site will be sufficient.

- Contingency is estimated by applying 5% on the mine mobile equipment fleet, and 20% on all other capital cost items.
- Cost for mine buildings such as truck shop, mine dry, fuel storage, assay lab, and storage facilities are included with the infrastructure costs.

### 21.2.9 WASTE AND WATER MANAGEMENT CAPITAL COST ESTIMATE

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

- Contractor mobilization and demobilization.
- Site preparation for the TSF embankment footprints, laydown areas, topsoil and unsuitables stockpiles including clearing and grubbing, wetland dewatering and excavation, select service road construction, placement of a wearing course on the laydown area, construction dewatering and sediment and erosion control Best Management Practices (BMPs). Topsoil stripping and removal of unsuitable material from these areas and the majority of service roads have been included by MMTS.
- Earthworks costs for both TSF embankments. The total earthworks costs are integrated between MMTS and Knight Piésold, with MMTS covering much of the material haulage costs from the open pit and borrow source.
- Diversions ditches, Boswell Lake diversion embankment and overflow channel.
- Tailings distribution and embankment seepage collection and recycle systems.
- TSF embankment monitoring instrumentation (piezometers and inclinometers).
- EPCM (7% of the total estimate, as provided by Tetra Tech).
- Indirects (10% of the total estimate, as provided by Tetra Tech).
- Contingency (25% of the total estimate, as provided by Tetra Tech).

Development of initial and sustaining capital costs for the waste and water management facilities necessitated assumptions of the geotechnical site conditions which must be verified.

The cost estimate was compiled using information from similar projects, engineering experience and unit rates built up using first principals based on standard contractor rates in BC. The unit rates assumed a labour rate of $93.50/h, as provided by Tetra Tech.

### 21.2.10 Capital Costs Exclusions

The following items have been excluded from the CAPEX:

- working or deferred capital
- financing costs
- refundable taxes and duties
- land acquisition
- currency fluctuations
- lost time due to severe weather conditions
- lost time due to force majeure
- additional costs for accelerated or decelerated deliveries of equipment, materials, or services resultant from a change in project schedule
- warehouse inventories, other than those supplied in initial fills
- Owner's costs, unless provided by the Owner
- any project sunk costs (studies, exploration programs, etc.)
- mine reclamation costs (included in financial model)
- mine closure costs (included in financial model)
- escalation costs
- community relations.

## 21.3 Operating Cost Estimate

### 21.3.1 Summary of Operating Costs

On site operating costs are estimated to be US$10.68/t of material milled including mining, processing, G&A, and plant services. The unit costs summarized in Table 21.2 are based on a production rate of 40,000 t/d, 365 d/a of operation.

**Table 21.2 Operating Cost Summary**

| Area | Unit Cost (CAD$/t milled) |
|---|---|
| Mining | 5.24 |
| Processing | 4.49 |
| Tailings | 0.04 |
| G&A | 0.59 |
| Off-site Costs (Including Royalty) | 0.42 |
| **Total Operating Cost** | **10.78** |

### 21.3.2 MINING OPERATING COST ESTIMATE

All mining costs in this section are provided in Q4 2012 Canadian dollars. The average mine operating cost is estimated to be CAD$1.64/t material mined or CAD$5.24/t mill feed over the producing life of mine. Costs incurred during the pre-production period are included as capital costs. The average operating cost for material moved is $1.59/t and this cost includes the material movement to and from the ROM stockpile.

The mining OPEX consists of all mining activities from the pit to the tipping point at the feed crusher and waste truck dumps. The cost for waste rock hauled to the tailings facility for embankment construction is also included in the mining OPEX. Mine and maintenance activities associated with loading, hauling, drilling, blasting, pit support, mine maintenance support, pit dewatering, general mine expense, and engineering are included. Table 21.3 shows the estimated LOM operating costs for each of the areas.

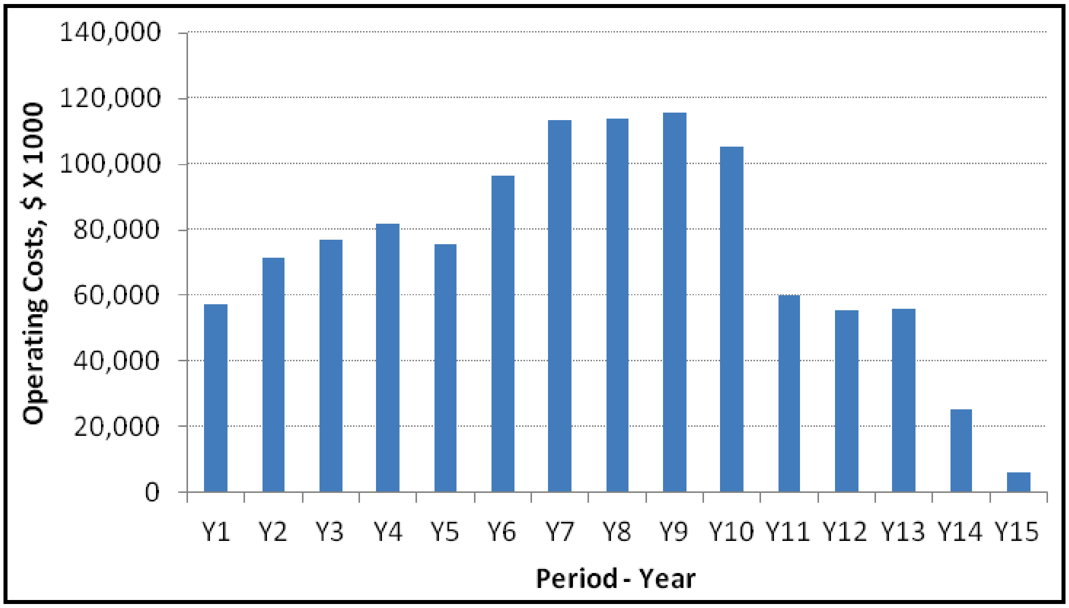
**Table 21.3 Mining Operating Cost Estimate (LOM)**

| Description | CAD$/t Mined | CAD$/t Mill Feed |
|---|---|---|
| Drilling | 0.11 | 0.35 |
| Blasting | 0.21 | 0.66 |
| Loading | 0.26 | 0.82 |
| Hauling | 0.75 | 2.41 |
| Pit Support | 0.17 | 0.53 |
| Mine Maintenance Support | 0.01 | 0.04 |
| Pit Dewatering | 0.02 | 0.07 |
| General Mine Expense | 0.06 | 0.19 |
| Engineering | 0.05 | 0.17 |
| **Total** | **1.64** | **5.24** |

Equipment productivities were calculated for the primary production equipment and applied against the annual production quantities to estimate equipment operating hours. Consumption rates for consumables and unit operating costs are applied to the equipment hours to calculate the total equipment operating costs for each period. The cost of parts and repairs are included in the operating costs for the major mining equipment. Major part replacements are based on manufacturer's recommendations and are expensed in the year in which they are forecasted.

The operating costs fluctuate annually and are a reflection of the total material mined and haulage distances. Balancing the waste quantities and the haulage destinations will smooth the operating costs and minimize fluctuations. Some smoothing is applied to the production schedule in this study. The spending profile is shown in Figure 21.1.

**Figure 21.1 Mine Operating Cost Profile**




The cost assumed for diesel is $1.10/L. Fuel consumption for explosives and the mine equipment fleet is estimated to average 18 ML/a during Years 1 to 5 and increases to 29 ML/a during Years 6 to 10, corresponding to the increase in material quantities mined. As waste material production decline over the last four year period, the annual fuel consumption is reduced to an average of 13 ML/a.

Explosives quantities are calculated using an estimated powder factor of 0.25 kg/bcm by a vendor from their experience at an existing nearby operation. Costs for explosives and blasting materials are from budgetary quotations provided the vendor and are based on delivery to the hole. The standard products are a 65/35 ratio of ammonium nitrate/fuel oil to emulsion for dry blastholes, and 30/70 ratio for wet blast-holes. For this study, it is assumed that dry holes will be achievable 75% of the time as a result of implementing a rigorous pit dewatering program. If dewatering is unsuccessful, emulsion explosives usage and blasting costs will increase substantially. It is estimated that the average annual explosives consumption will be approximately 10 Mkg during the first five operating years, and 17 Mkg during Years 6 to 10.

Salaries and hourly labour rates are estimates from other studies and mine operations in BC. The labour rates are applied to the operating and maintenance workforce generated from the equipment fleet to determine the total hourly labour cost. Salaries are applied to the total number of mine operations and engineering personnel to arrive at the salaried cost.

Dewatering costs are based on a preliminary depressurization plan provided by BGC. The water from the wells will be discharged directly to the plant site and assumed to be suitable for processing. Costs include allowances for installation of horizontal and vertical dewatering wells, pumps, and pump maintenance. Costs for pipe installation, maintenance, and power to operate the pumps are estimated with the process costs. Costs for in-pit pumping are included with pit support. The labour cost for the pump crew is included with mine expense.

#### MINING LABOUR

The mine workforce estimate is summarized in Table 21.4. The hourly labour workforce reflects the annual quantity of material mined. The staff positions are less variable with the production rate and are consistent throughout the LOM except for a reduction over the last five years. During the first five years the annual mine labour count averages 168. It increases to an average of 228 during the next five-year period when the average waste production rate is more than doubled. When the strip ratio is reduced after ten years, the mine labour force is reduced significantly.

**Table 21.4 Mining Operations Labour – Average for Periods**

| | Years 1 to 5 | Years 6 to 10 | Years 11 to 14 |
|---|---|---|---|
| Equipment Operators | 102 | 146 | 70.0 |
| Mine Maintenance | 35 | 51 | 25.0 |
| **Subtotal** | **137** | **197** | **95.0** |
| Mine Superintendent | 1 | 1 | 1.0 |
| Mine & Maintenance General Foremen | 2 | 2 | 1.5 |
| Shift Foremen/Team Leaders | 9 | 9 | 7.0 |
| Trainers | 1 | 1 | 0.5 |
| Maintenance Planners | 1 | 1 | 0.7 |
| Clerks | 2 | 2 | 1.5 |
| **Subtotal** | **16** | **16** | **12.2** |
| Mine Technical | | | |
| Chief Engineer | 1 | 1 | 1.0 |
| Geologists | 3 | 3 | 2.0 |
| Mine Engineers | 6 | 6 | 4.0 |
| Technicians/Surveyors | 5 | 5 | 4.0 |
| **Subtotal** | **15** | **15** | **13.0** |
| **Total Salaried Staff** | **31** | **31** | **23.0** |
| **Total Mine Workforce** | **168** | **228** | **118.0** |

### *21.3.3 PROCESS OPERATING COST ESTIMATE AND G&A COSTS*

The process operating costs for the Spanish Mountain concentrator includes crushing, grinding, flotation, concentrate regrind, CIL, electrowinning and refining to produce a saleable gold doré, along with tailings disposal. Estimated G&A costs are included in the OPEX.

### OPERATING COST SUMMARY

Table 21.5 summarizes the overall estimated cost summary for the processing facility and the G&A costs, based on a 40,000 t/d mill operating 365 d/a, at a 92% availability. The operating costs are based on the operating plant complement required to run and maintain the plant facilities. The manpower complement is based on four crews working an equal rotation time of two 12-hour shifts per day.

The annual operating costs for the process facilities and tailings handling sections is estimated to be CAD$59.3 million, or CAD$4.06/t of plant feed material treated at the processing rate of 40,000 t/d. An incremental increase in the operating cost distributed to G&A is estimated to be CAD$8.56 million, or CAD$0.59/t of material treated.

**Table 21.5 Operating Cost Summary**

| Description | Personnel | Annual Cost (CAD$) | Unit Cost (CAD$/t Treated) |
|---|---|---|---|
| **Complement** | | | |
| Operations Staff | 5 | 618,750 | 0.042 |
| Operations Labour | 41 | 3,513,800 | 0.241 |
| Maintenance Labour | 15 | 1,434,590 | 0.098 |
| Laboratories and Quality Control | 10 | 744,600 | 0.051 |
| **Subtotal – Labour** | **71** | **6,311,740** | **0.432** |
| **Plant Supplies** | | | |
| Operating Supplies | - | 42,184,593 | 2.889 |
| Maintenance Supplies | - | 3,499,483 | 0.240 |
| Power Supply | - | 13,619,556 | 0.933 |
| **Subtotal – Power and Supply** | **-** | **59,303,633** | **4.062** |
| **Total Process Operating Costs** | **71** | **65,615,373** | **4.494** |
| G&A Staff | 29 | 2,541,250 | 0.174 |
| G&A Expenses | - | 5,950,000 | 0.408 |
| G&A Power Cost | - | 64,200 | 0.004 |
| **Total G&A Costs** | **29** | **8,555,450** | **0.586** |

The annual operating cost estimate includes the following:

- the staffing and maintenance manpower base salaries is based on information available from similar projects conducted by Tetra Tech, the burden value of 25% is based on information as supplied by SMG.
- power consumption is based on the estimated power usage with the cost of power provided by Tetra Tech based on rates supplied by BC Hydro
- reagent consumption values, and the cost of reagents, has been based on prices from suppliers

- estimated maintenance costs
- estimated cost of plant supplies
- the staffing and manpower complement based on typical operating mills as revised by Tetra Tech with input from SMG.

### MANPOWER

Table 21.6 and Table 21.7 outlines the estimated operating and maintenance manpower requirements for the process plant; Table 21.8 shows the G&A complement.

**Table 21.6 Process Plant Manpower Requirements**

| Description | Manpower | Loaded Annual Salary (CAD$) | Annual Cost Payroll (CAD$) | Unit Cost (CAD$/t Treated) |
|---|---|---|---|---|
| **Average Benefit Rate/Burden** | - | 25% | - | - |
| **Staff** | | | | |
| Mill Superintendent (Plant Manager) | 1 | 206,250 | 206,250 | 0.014 |
| Chief Metallurgist | 0 | 187,500 | 0 | 0.000 |
| General Foreman | 1 | 137,500 | 137,500 | 0.009 |
| Chief Assayer (Chief Chemist) | 1 | 143,750 | 143,750 | 0.010 |
| Senior Chemist | 0 | 106,250 | 0 | 0.000 |
| Mill Clerk/ Planner | 2 | 68,750 | 137,500 | 0.009 |
| **Subtotal 1** | **5** | **-** | **618,750** | **0.042** |
| **Operations** | | | | |
| Mill Foreman (Shift Supervisor) | 4 | 118,750 | 475,000 | 0.033 |
| Senior Metallurgists | 1 | 143,750 | 143,750 | 0.010 |
| Junior Metallurgists | 1 | 93,750 | 93,750 | 0.006 |
| Plant Trainer | 1 | 93,750 | 93,750 | 0.006 |
| Primary Crusher Operators | 4 | 76,650 | 306,600 | 0.021 |
| Control Room Operators | 4 | 98,550 | 394,200 | 0.027 |
| Grinding Operators | 4 | 82,125 | 328,500 | 0.023 |
| Flotation Operators | 4 | 90,338 | 361,350 | 0.025 |
| Gravity Circuit Operators including Intensive Cyanidation | 2 | 78,000 | 156,000 | 0.011 |
| CIL Operators & Refinery | 6 | 83,200 | 499,200 | 0.034 |
| Reagent Operators | 2 | 76,650 | 153,300 | 0.011 |
| Tailings Handling | 4 | 76,650 | 306,600 | 0.021 |
| Labourer/Trainee | 4 | 62,963 | 251,850 | 0.017 |
| **Subtotal 2** | **41** | **-** | **3,513,800** | **0.241** |

*table continues…*

| Description | Manpower | Loaded Annual Salary (CAD$) | Annual Cost Payroll (CAD$) | Unit Cost (CAD$/t Treated) |
|---|---|---|---|---|
| **Metallurgical Laboratory and Quality Control** | | | | |
| Metallurgical Technicians | 0 | 78,000 | 0 | 0.000 |
| Assayer | 6 | 82,125 | 492,750 | 0.034 |
| Sample Bucker | 4 | 62,963 | 251,850 | 0.017 |
| **Subtotal 3** | **10** | **-** | **744,600** | **0.051** |
| **Total Process Plant Manpower** | **56** | **-** | **4,877,150** | **0.344** |

**Table 21.7 Process Plant Maintenance Manpower Requirements**

| Description | Manpower | Loaded Annual Salary (CAD$) | Annual Cost Payroll (CAD$) | Unit Cost (CAD$/t Treated) |
|---|---|---|---|---|
| **Average Benefit Rate/Burden** | **-** | **25%** | **-** | **-** |
| Mill Maintenance Foremen | 1 | 143,750 | 143,750 | 0.010 |
| Mill Maintenance Clerk/Planner | 2 | 86,970 | 173,940 | 0.012 |
| Millwrights-Journeyman | 4 | 95,813 | 383,250 | 0.026 |
| Welders-Journeyman | 2 | 95,813 | 191,625 | 0.013 |
| Apprentices (Level 2) | 2 | 79,388 | 158,775 | 0.011 |
| Electrician | 2 | 95,813 | 191,625 | 0.013 |
| Instrument Technicians | 2 | 95,813 | 191,625 | 0.013 |
| **Total Maintenance Manpower** | **15** | | **1,434,590** | **0.098** |

**Table 21.8 G&A Manpower**

| Description | Manpower | Annual Cost/ Employee (CAD$) | Annual Cost Payroll (CAD$) | Unit Cost (CAD$/t Treated) |
|---|---|---|---|---|
| **Average Benefit Rate/Burden** | - | 25% | - | - |
| **General Administration** | | | | |
| General Manager | 1 | 256,250 | 256,250 | 0.018 |
| Secretary | 1 | 62,500 | 62,500 | 0.004 |
| Administration Manager | 1 | 100,000 | 100,000 | 0.007 |
| HSEC Manager | 1 | 112,500 | 112,500 | 0.008 |
| IT Services Manager | 0 | 87,500 | 0 | 0.000 |
| IT Technologists | 2 | 75,000 | 150,000 | 0.010 |

*table continues…*

| Description | Manpower | Annual Cost/ Employee (CAD$) | Annual Cost Payroll (CAD$) | Unit Cost (CAD$/t Treated) |
|---|---|---|---|---|
| **Accounting** | | | | |
| Chief Accountant | 1 | 112,500 | 112,500 | 0.008 |
| Payroll Clerk | 1 | 75,000 | 75,000 | 0.005 |
| Accounting Clerk | 1 | 50,000 | 50,000 | 0.003 |
| **Purchasing/Warehouse** | | | | |
| Chief Purchaser | 1 | 112,500 | 112,500 | 0.008 |
| Assistant Purchaser | 1 | 87,500 | 87,500 | 0.006 |
| Warehouse Shipper & Receiver | 2 | 68,750 | 137,500 | 0.009 |
| Janitorial | 2 | 67,500 | 135,000 | 0.009 |
| Gate Security/First Aid | 4 | 75,000 | 300,000 | 0.021 |
| **Human Resources and Training** | | | | |
| Human Resources Manager | 1 | 112,500 | 112,500 | 0.008 |
| Human Resources Clerk | 0 | 75,000 | 0 | 0.000 |
| **Health and Safety** | | | | |
| Health and Safety Co-ordinator | 1 | 100,000 | 100,000 | 0.007 |
| Safety and Training Officer | 0 | 91,250 | 0 | 0.000 |
| First Aid Workers | 0 | 78,750 | 0 | 0.000 |
| **Environmental** | | | | |
| Environmental and Community Relations Manager | 0 | 112,500 | 0 | 0.000 |
| Environmental Technician | 2 | 87,500 | 175,000 | 0.012 |
| **Site Services** | | | | |
| Site Services General Foreman | 1 | 87,500 | 87,500 | 0.006 |
| Site Yard Labourers | 2 | 62,500 | 125,000 | 0.009 |
| Site Electrician | 1 | 75,000 | 75,000 | 0.005 |
| Site Mechanic | 1 | 87,500 | 87,500 | 0.006 |
| Site Carpenter | 1 | 87,500 | 87,500 | 0.006 |
| **Total G&A Manpower** | **29** | **-** | **2,541,250** | **0.174** |

### POWER AND SUPPLIES

Table 21.9 through Table 21.13 show the operating cost details relating to power and supplies, as well as the G&A expenses.

**Table 21.9 Power Supply Required for Process**

| Area | kWh/a | Unit Cost (CAD$/kWh) | Total Cost (CAD$/a) | Unit Cost (CAD$/t Treated) |
|---|---|---|---|---|
| **Total Plant Power Supply** | **318,213,925** | **0.043** | **13,619,556** | **0.933** |
| **G&A Power Supply** | **1,500,000** | **0.043** | **64,200** | **0.004** |

**Table 21.10 Maintenance Supplies Allowance**

| Area | Estimate Basis | Total Cost (CAD$/a) | Unit Cost (CAD$/t Treated) |
|---|---|---|---|
| Primary Crusher and Conveyors | 2.5% of Main Equipment Capital Cost | 729,974 | 0.050 |
| Grinding and Gravity Area | | 1,212,091 | 0.083 |
| Flotation Area | | 247,162 | 0.017 |
| CIL Area | | 324,829 | 0.022 |
| Carbon Circuit, Electrowinning and Smelting | | 58,275 | 0.004 |
| Detoxification | | 11,368 | 0.001 |
| Tailings Handling | | 12,500 | 0.001 |
| Reagents | | 65,835 | 0.005 |
| Assaying | Estimate | 150,000 | 0.010 |
| Miscellaneous Plant Supplies | Estimate | 250,000 | 0.017 |
| Miscellaneous Building Complex Supplies | 0.5% of Capital Cost | 437,450 | 0.030 |
| **Total Maintenance Supplies** | **-** | **3,499,483** | **0.240** |

**Table 21.11 Plant Operating Supplies**

| Supplies | Consumption (kg/t Treated) | Unit Cost (CAD$/kg) | Total Cost (CAD$/a) | Unit Cost (CAD$/t Treated) |
|---|---|---|---|---|
| Gyratory Crusher Liners | 1 set/year | 400,000/ all spares | 400,000 | 0.027 |
| Cone Crusher Liners | 1 set/ crusher/year | 150,000/set | 300,000 | 0.021 |
| Screen Panels | 2 sets/ screen/year | 10,000/set | 180,000 | 0.012 |
| SAG Mill Media | 0.283 | 1.010 | 4,173,118 | 0.286 |
| SAG Mill Liners | 1 set/year | 1,630,000/set | 1,630,000 | 0.112 |
| Ball Mill Media | 0.394 | 0.940 | 5,410,687 | 0.371 |
| Ball Mill Liners | 1 set/year | 774,000/set | 774,000 | 0.053 |
| Isamill Regrind Media | 0.005 | 5.000 | 368,854 | 0.025 |
| Isamill Spare Grinding Discs | 6 sets/year | 14,000/set | 84,000 | 0.006 |
| Isamill Spare Liners | 1 set/year | 134,000/set | 134,000 | 0.009 |
| Conveyors | - | - | 0 | 0.000 |
| **Subtotal – Plant** | **-** | **-** | **13,454,659** | **0.922** |
| PAX | 0.090 | 2.600 | 3,432,343 | 0.235 |
| MIBC | 0.075 | 2.750 | 3,011,250 | 0.206 |
| Lime | 0.055 | 0.230 | 184,690 | 0.013 |
| CMC(SD200) | 0.122 | 4.500 | 8,021,970 | 0.549 |
| Sodium Cyanide | 0.234 | 3.000 | 10,249,200 | 0.702 |
| Copper Sulphate | 0.0014 | 2.950 | 59,975 | 0.004 |
| SMBS | 0.022 | 0.890 | 285,868 | 0.020 |
| Sodium Hydroxide | 0.002 | 1.000 | 24,382 | 0.002 |

*table continues…*

| Supplies | Consumption (kg/t Treated) | Unit Cost (CAD$/kg) | Total Cost (CAD$/a) | Unit Cost (CAD$/t Treated) |
|---|---|---|---|---|
| Hydrochloric Acid | 0.0003 | 0.500 | 2,208 | 0.000 |
| Leach-Aid | 0.0001 | 35.33 | 25,793 | 0.002 |
| Flocculant | 0.001 | 4.250 | 45,297 | 0.003 |
| Flux | 0.001 | - | - | - |
| $SiO_2$ (%) | 30 | 1.000 | 2,738 | 0.0002 |
| Borax (%) | 40 | 1.900 | 6,935 | 0.0005 |
| Niter (%) | 10 | 2.200 | 2,008 | 0.0001 |
| Soda Ash (%) | 20 | 0.700 | 1,278 | 0.0001 |
| Carbon | 0.050 | 3.800 | 2,774,000 | 0.190 |
| **Subtotal – Reagents** | **-** | **-** | **28,129,934** | **1.927** |
| Laboratory Supplies | allowance | 250,000 | 250,000 | 0.017 |
| **Subtotal – Laboratory Supplies** | **-** | - | **250,000** | **0.017** |
| Vehicle Operation | 2.5% of cost | estimate | 100,000 | 0.007 |
| Miscellaneous | allowance | - | 250,000 | 0.017 |
| **Subtotal – Others** | **-** | - | **350,000** | **0.024** |
| **Total Operating Supplies** | **-** | - | **42,184,593** | **2.889** |

**Table 21.12 G&A Expenses**

| | Total Cost (CAD$/a) | Unit Cost (CAD$/t Treated) |
|---|---|---|
| Corporate Office Expenses | 150,000 | 0.010 |
| Office Supplies | 100,000 | 0.007 |
| Professional Associations | 50,000 | 0.003 |
| Consultants | 100,000 | 0.007 |
| Insurance | 1,200,000 | 0.082 |
| Legal Services | 100,000 | 0.007 |
| Regulatory Compliance/Audit | 100,000 | 0.007 |
| Travel and Expenses | 80,000 | 0.005 |
| Communications: Telephone/Fax/Internet | 250,000 | 0.017 |
| Computer and IT Services and Supplies | 120,000 | 0.008 |
| Community Public Relations and Donations | 200,000 | 0.014 |
| Recruitment | 150,000 | 0.010 |
| Training | 250,000 | 0.017 |
| Safety and Training Supplies | 300,000 | 0.021 |
| Safety Incentives | 250,000 | 0.017 |
| Medical Service/First Aid | 250,000 | 0.017 |
| Security Supplies | 200,000 | 0.014 |
| Environmental Supplies | 400,000 | 0.027 |
| Purchasing and Logistics | 150,000 | 0.010 |
| External Assays/Testings | 250,000 | 0.017 |

*table continues…*

| | Total Cost (CAD$/a) | Unit Cost (CAD$/t Treated) |
|---|---|---|
| Janitorial | 150,000 | 0.010 |
| Powerline Maintenance | 300,000 | 0.021 |
| Road Maintenance | 100,000 | 0.007 |
| Crew Transportation | 500,000 | 0.034 |
| Miscellaneous | 250,000 | 0.017 |
| **Total G&A Expenses** | **5,950,000** | **0.408** |

**Table 21.13 First Fills – Capital Cost**

| Supplies | Source | Total Cost (CAD$) |
|---|---|---|
| Reagents | Estimate | 2,344,161 |
| Liners and Grinding Media | Estimate | 1,219,388 |

*General Discussion*

- Table 21.9 gives the annual power requirements for the process plant unit operations as based on the estimated power usage. The unit cost of power was calculated to be CAD$0.043/kWh and equates to CAD$0.933/t of material processed. The power requirements for the G&A portion is listed separately in Table 21.9 and amounts to CAD$0.004/t treated.
- The overall anticipated costs for annual maintenance supplies are shown in Table 21.10. The plant cost of maintenance supplies has been calculated to be CAD$3.50 million, or CAD$0.240/t of material treated in the plant.
- The annual estimated plant operating supplies requirements are provided in Table 21.11. The operating supply costs have also been estimated for the various sections of the process plant. The costs of the grinding media and reagents have been included in this table. The total cost of operating supplies for the main plant has been determined to be CAD$42.18 million, or CAD$2.89/t of material processed.
- Table 21.12 details the annual G&A expenses and does not include any contingency. The unit G&A expenses operating cost have been calculated to be CAD$5.95 million, or CAD$0.41/t of material processed.
- The process section “first fills” requirement is provided in Table 21.13, although this is generally considered to be a capital cost item.

*Assumptions Used in Power and Supplies Requirements*

- The estimated power requirements for the process sections are given as a total value in Table 21.9. The power value records the anticipated annual operating power usage and not the installed power for the equipment. The unit electrical cost of CAD$0.043/kWh was calculated by Tetra Tech based on rates supplied by BC Hydro.

- The costs of the plant operating supplies includes the reagent costs for the reagent consumption values as given in the process design criteria. The reagent usage for PAX and CMC has been assumed to be 66% of the test work consumption, to account for the scale-up from laboratory test work to commercial scale operations. The grinding media costs are based on the calculated consumption values, and vendor estimates.
- The maintenance supplies costs are estimated values and are reflected as an allowance as based on a percentage of the CAPEX. Similarly, the G&A expenses have been based on estimated allowance values.
- First fills costs are estimates only, based on the yearly consumption values used in the OPEX. The reagent and steel allotment is an allowance for a one month's supply only, since delivery interruptions are unlikely given the location of the mill site.

### 21.3.4 WASTE AND WATER MANAGEMENT OPERATING COSTS

Operating costs for the waste and water management facilities were completed for the tailings distribution and embankment seepage collection and recycle systems using a unit rate for power of CAD$43/MWh.

### 21.3.5 FISHERIES COMPENSATION AND ENVIRONMENTAL MONITORING

#### WASTE AND WATER MANAGEMENT FACILITIES

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

Development of the TSF will directly affect Nina Lake, the mainstem of Cedar Creek and several unnamed tributaries. It will also indirectly affect the lower reaches of Cedar Creek. Fish habitat compensation ratios were calculated as 2:1, with assumed unit area capital costs of CAD$150,000/ha for instream habitat and CAD$50,000/ha for riparian habitat.

Based on these unit area costs, fisheries compensation is anticipated to cost approximately CAD$10 million; CAD$1 million per year has been allocated for environmental monitoring.

# 22.0 ECONOMIC ANALYSIS

**A PEA should not be considered a prefeasibility or feasibility study, as the economics and technical viability of the Project have not been demonstrated at this time. The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Furthermore, there is no certainty that the conclusions or results as reported in the PEA will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.**

An economic evaluation of the Project was carried out by Tetra Tech incorporating all the relevant capital, operating, working, sustaining costs, and royalties (1.5% of NSR). The evaluation was based on a pre-tax financial model and was calculated in US dollars. For the 15-year mine life and 206 Mt resource inventory, the following pre-tax financial parameters were calculated using the base case gold price:

- 15% IRR
- 4.4-year payback on US$756 million capital
- US$454 million NPV at 5% discount value.

SMG commissioned PwC in Vancouver, BC to prepare a tax model for the post-tax economic evaluation of the Project with the inclusion of applicable income and mining taxes. (see Section 22.4).

The following post-tax financial parameters were calculated:

- 12% IRR
- 4.7-year payback on US$756 million capital
- US$291 million NPV at an 5% discount rate.

The gold and silver prices used for the base case are US$1,462/oz and US$28.13/oz respectively, using the three-year trailing average (as of November 1, 2012). The base case exchange rate was set at US$0.9905:CAD$1.00, also using the three-year trailing average.

Sensitivity analyses were developed to evaluate the Project economics.

## 22.1 PRE-TAX MODEL

### 22.1.1 METAL PRODUCTION IN FINANCIAL MODEL

The revenues projected in the cash flow model were based on the average metal values indicated in Table 22.1.

**Table 22.1 Metal Production from Spanish Mountain Gold Project**

| | Years 1 to 3 | LOM |
|---|---|---|
| Total Tonnes to Mill ('000) | 38,839 | 205,896 |
| Annual Tonnes to Mill ('000) | 12,946 | 13,726 |
| **Average Grade** | | |
| Gold (g/t) | 0.697 | 0.481 |
| Silver (g/t) | 0.675 | 0.673 |
| **Total Production** | | |
| Gold ('000 oz) | 786 | 2,799 |
| Silver ('000 oz) | 211 | 1,114 |
| **Average Annual Production** | | |
| Gold ('000 oz) | 262 | 187 |
| Silver ('000 oz) | 70 | 74 |

### 22.1.2 FINANCIAL EVALUATIONS – NPV AND IRR

The production schedule has been incorporated into the 100% equity pre-tax financial model to develop annual recovered metal production from the relationships of tonnage processed, head grades, and recoveries.

Gold revenues were calculated based on market prices. Unit operating costs for mining, processing, site services, G&A, and off-site charges (smelting, transportation, and royalties) areas were applied to annual milled tonnages to determine the overall operating cost, which was deducted from the revenues to derive annual operating cash flow (net revenue).

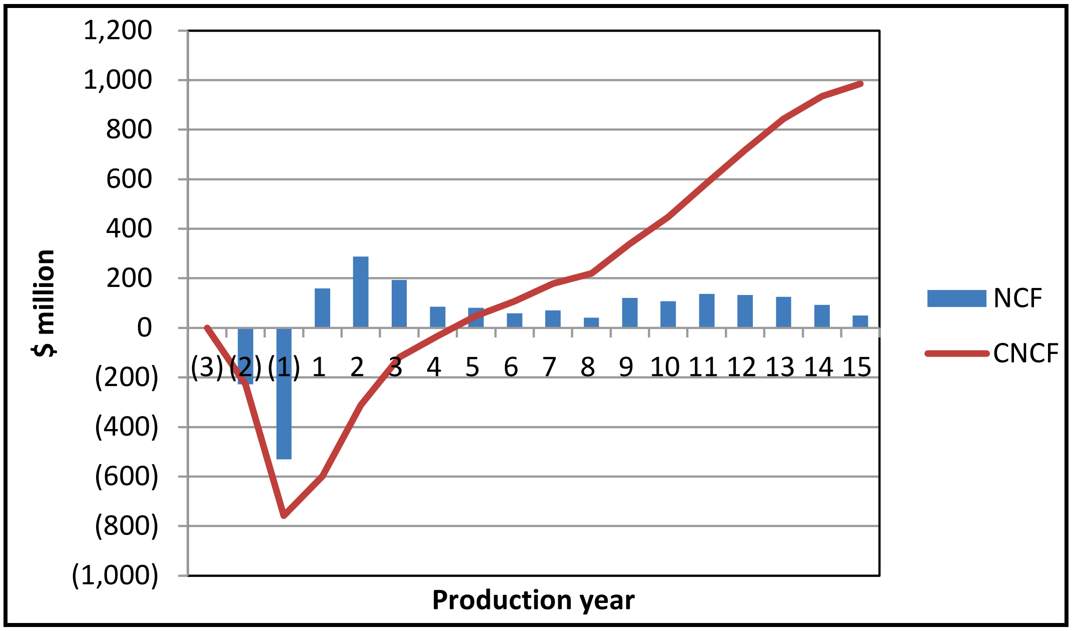
Initial and sustaining capital costs have been incorporated on a year-by-year basis over the mine life and deducted from the net revenue to determine the net cash flow before taxes. Initial capital expenditures include costs accumulated prior to first production of gold doré; sustaining capital includes expenditures for mining and processing additions, replacement of equipment, tailing embankment construction, water treatment, and environmental/closure costs.

The financial analysis used three months of the on-site operating cost as working capital cost. The working capital is recovered at the end of the mine life.

Reclamation costs were estimated at US$17 million.

The undiscounted annual cash flows are illustrated in Figure 22.1.

**Figure 22.1 Pre-tax Undiscounted Annual and Cumulative Cash Flow**




Notes: NCF = net cash flow; CNCF = cumulative net cash flow

## 22.2 METAL PRICE SCENARIOS

Tetra Tech evaluated the base case, as well as two additional metal price scenarios using a lower price case and the spot gold and silver price on November 1, 2012.

The pre-tax financial model was established on a 100% equity basis, excluding debt financing and loan interest charges. The financial outcomes have been tabulated for NPV, IRR, payback of capital, and cost per ounce of gold. Discount rates of 0%, 5% and 8% were applied to all scenarios. The results are presented in Table 22.2.

**Table 22.2 Summary of Pre-tax NPV, IRR and Payback by Metal Price Scenario**

| | Unit | Base Case | Lower Price Case | Spot Price Case |
|---|---|---|---|---|
| **Metal Prices** | | | | |
| Gold | US$/oz | 1,462.00 | 1,350.00 | 1,716.00 |
| Silver | US$/oz | 28.13 | 20.00 | 32.66 |
| Exchange Rate | US$:CAD$ | 0.99 | 1.00 | 1.00 |
| **Economic Results** | | | | |
| NPV (at 0%) | US$ million | 985 | 640 | 1,650 |
| NPV (at 5%) | US$ million | 454 | 227 | 887 |
| NPV (at 8%) | US$ million | 261 | 79 | 605 |
| IRR | % | 15.00% | 10.19% | 23.04% |
| Payback | years | 4.40 | 7.45 | 2.75 |
| Cash Cost/oz Au | US$/oz | 774 | 782 | 786 |
| Total Cost/oz Au | US$/oz | 1,110 | 1,121 | 1,127 |

Note: Total costs per ounce include all start-up capital, sustaining capital, and reclamation/closure costs.

## 22.3 SENSITIVITY ANALYSIS

Sensitivity analyses were carried out on the following parameters:

- gold price
- silver price
- exchange rate
- initial capital expenditure
- on-site operating costs.

The analyses are presented graphically as financial outcomes in terms of NPV and IRR. The Project NPV is most sensitive to gold price and operating costs followed by exchange rate and initial capital with silver price having the least impact. The Project IRR is most sensitive to exchange rate and operating costs followed by gold price and initial capital with silver price having the least impact. The NPV and IRR sensitivities are shown in Figure 22.2 and Figure 22.3.

**Figure 22.2 Sensitivity Analysis of Base Case Pre-tax NPV at 5% Discount Rate**

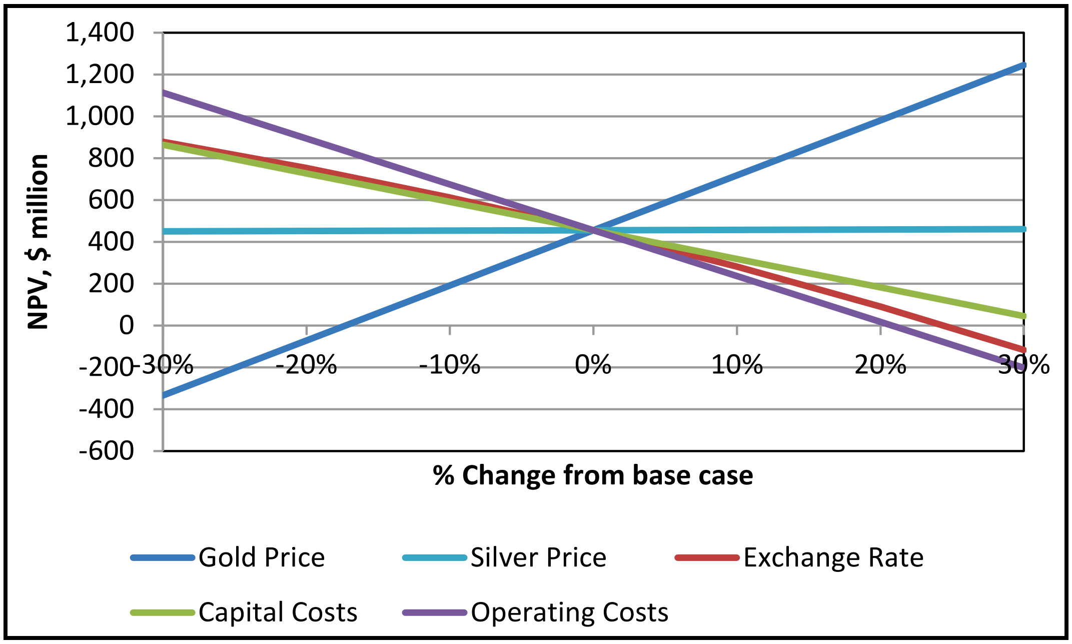
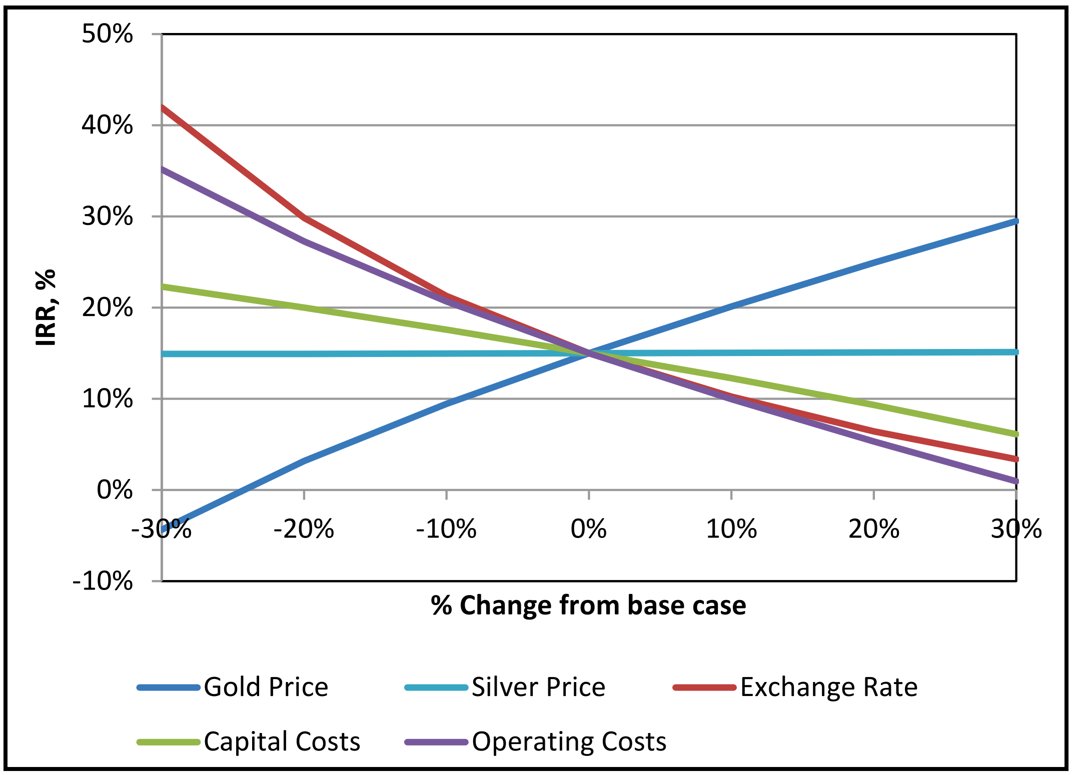



**Figure 22.3 Sensitivity Analysis of Pre-tax Base Case IRR**



## 22.4 Post-tax Financial Analysis

SMG commissioned PwC in Vancouver, BC to prepare a tax model for the post-tax economic evaluation of the Project with the inclusion of applicable income and mining taxes.

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

**Canadian Federal and BC Provincial Income Tax Regime**

Federal and BC provincial income taxes are calculated using the currently enacted corporate rates of 15% for federal and 10% for BC.

For both federal and provincial income tax purposes, capital expenditures are accumulated in pools that can be deducted against mine income at different rates, depending on the type of capital expenditure.

Resource property acquisition costs and the costs of mine shafts, main haulage ways and other underground workings are accumulated in the Canadian Development Expense (CDE) pool. The CDE is amortized against income at 30% on a declining balance basis.

Most other pre-production mine development expenditures are accumulated in the Canadian Exploration Expense (CEE) pool. The CEE is generally amortized at 100%, to the extent of taxable income from the mine.

Fixed assets purchased prior to the commencement of production are accumulated in an Undepreciated Capital Cost pool (Class 41(a)) and are generally amortized at 100%, to the extent of taxable income from the mine.

Fixed assets purchased after the commencement of production, to the extent that the costs exceed 5% of gross revenue from the mine for the year, are accumulated in Class 41(a.1) and are also amortized at 100%, to the extent of taxable income from the mine.

All other fixed assets purchased after the commencement of production are accumulated in Class 41(b) and are amortized at 25% on a declining balance basis.

**BC Mineral Tax Regime**

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

The 2% tax is assessed on "net current proceeds", which is defined as gross revenue from the mine less mine operating expenditures. Hedging income and losses, royalties and financing costs are excluded from operating expenditures. The 2% tax is accumulated in a Cumulative Tax Credit Account (CTCA) and is fully creditable against the 13% tax.

All capital expenditures, both mine development costs and fixed asset purchases, are accumulated in the Cumulative Expenditures Account (CEA), which is amortized at 100% against the 13% tax.

The 13% tax is assessed on "net revenue", which is defined as gross revenue from the mine, less mine operating expenditures, less any accumulated CEA balance. As such, the 13% tax is not assessed until all pre-production capital expenditures have been amortized.

A "new mine allowance" is provided for capital costs of new mines to encourage mine investment in BC. This allowance provides that 133% of capital expenditures incurred prior to commencement of production are included in the CEA. Under current legislation, the provision for the new mine allowance is scheduled to expire on January 1, 2016. The post-tax model is calculated on the assumption that the mine allowance will be continued.

Notional interest of 125% of the prevailing federal bank rate is calculated annually on any unused CEA and CTCA balances and is added to these pools.

The BC Mineral Tax is deductible for federal and provincial income tax purposes.

At the long-term metal prices used for this study, total estimated taxes payable on Spanish Mountain profits are US$290.57 million over the 15-year mine life. The components of the various taxes that will be payable are shown in Table 22.3.

**Table 22.3 Components of the Various Taxes**

| Tax Component | LOM Amount (US$ million) |
|---|---|
| Corporate Tax (Federal) | 128.49 |
| Corporate Tax (Provincial) | 85.66 |
| Less Investment Tax Credit (ITC) | (2.52) |
| Provincial Resource Tax | 78.95 |
| **Total Taxes** | **290.57** |

Base case post-tax financial results are summarized in Table 22.4.

**Table 22.4 Summary of Post-tax Financial Results**

| Description | Value |
|---|---|
| Gold Price (US$/oz) | 1,462.00 |
| Silver Price (US$/oz) | 28.13 |
| Exchange Rate (CAD$/US$) | 0.9905 |
| Net Cash Flow (US$ million) | 694 |
| Discounted Cash Flow NPV (US$ million) at 5% | 291 |

*table continues…*

| Description | Value |
|---|---|
| Discounted Cash Flow NPV (US$ million) at 8% | 142 |
| Payback (years from start of mill operations) | 4.7 |
| IRR (%) | 12.31 |

## 22.5 ROYALTIES

The royalty consists of a 1.5% NSR.

## 22.6 SMELTER TERMS

The following refining terms were applied in the financial analysis:

- gold – pay 99.8% of content less a refining charge of US$1.00/accountable ounce.
- silver – pay 90.0% of content less a refining charge of US$0.60/accountable ounce.

## 22.7 TRANSPORTATION LOGISTICS

Transportation cost of US$1.00/oz is applied to both gold and silver produced.

### 22.7.1 INSURANCE

An insurance rate of 0.15% was applied to the provisional invoice value of gold and silver.

# 23.0 ADJACENT PROPERTIES

The following section is derived directly from the report "Technical Report on an Updated Mineral Resource Estimate on the Spanish Mountain Gold Deposit", by G. Giroux and A. Koffyberg and dated August 31, 2012. Minor changes have been made for report consistency.

The Property is located in an area that has seen active past exploration and mining activity for alkaline porphyry copper-gold deposits. Currently, the most advanced property in the area is Imperial Metals' Mount Polley Mine, which is an alkalic porphyry copper-gold deposit located about 15 km to the west. As of March 2012, the deposit has a measured and indicated resource of 361.14 Mt of 0.284% copper, 0.297 g/t gold and 0.846 g/t silver (Imperial Metals website).

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

Various placer properties and operations on placer leases exist in and around the Likely area. Very little public information is available about the gold content in the placer deposits.

# 24.0 OTHER RELEVANT DATA AND INFORMATION

## 24.1 AGREEMENTS

In March 2011, a protocol agreement was signed with the Williams Lake Indian Band (WLIB). Under this agreement, SMG recognizes and respects WLIB's asserted aboriginal rights and title in the area of the Project; and the WLIB recognizes and respects SMG's rights and interests in the exploration and development of the Project.

In March 2012, a protocol agreement was signed with the Soda Creek Indian Band (SCIB). Similar to the agreement with the WLIB, rights, title and interests are respected by both parties. The agreement also provides capacity support to the SCIB for its ongoing involvement in the Project as well as training, employment and business opportunities.

A cooperation agreement was signed with the Lhtako Dene Nation (Lhtako Dene) on September 12, 2012. Under this agreement, SMG recognizes and respects Lhtako Dene's asserted aboriginal rights and title in the area of the Project and the Lhtako Dene recognizes and respects SMG's rights and interest in the exploration and development of the project. The agreement reflects a commitment from both parties for continued engagement in a respectful and collaborative manner.

Cedar Point Provincial Park is a small 8 ha Class C park, located where Cedar Creek enters Quesnel Lake (Figure 4.4). Part of the park underlies a small part of mineral claim 517485.

## 24.2 PROJECT SCHEDULE

### 24.2.1 INTRODUCTION

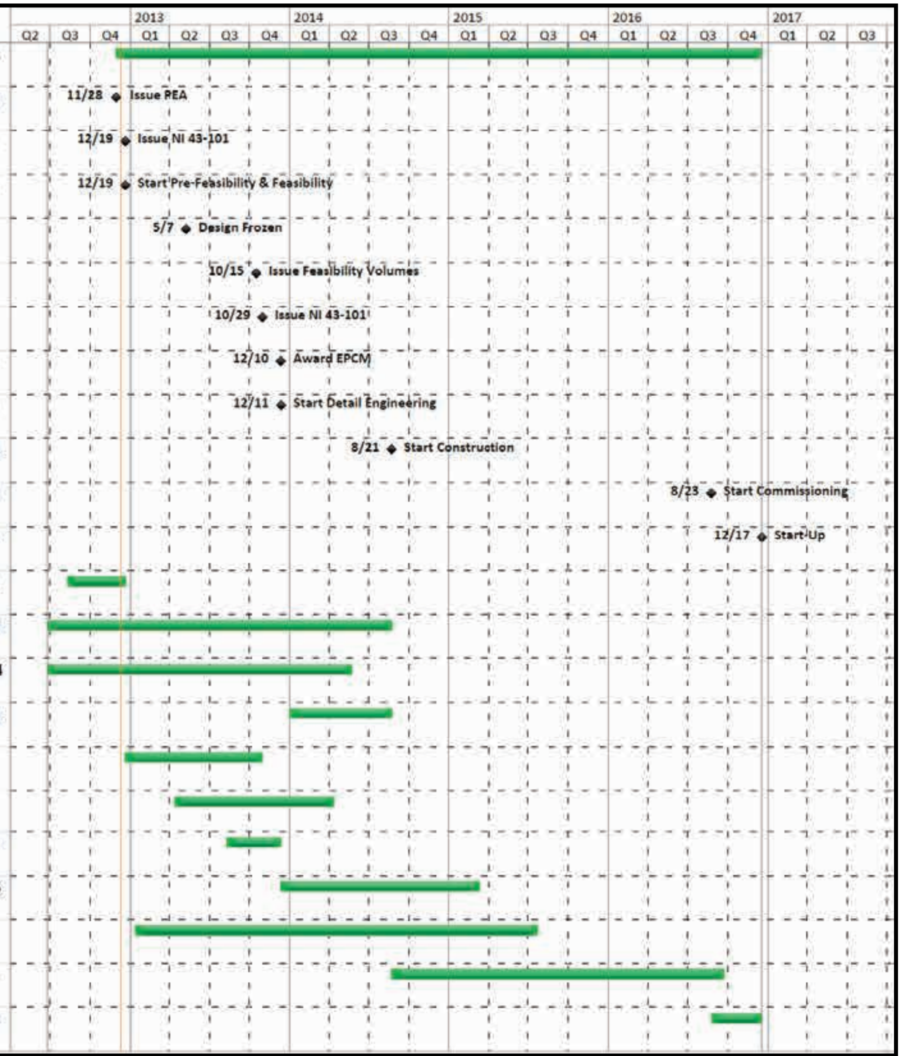
The Project will be designed and constructed to industry and regulatory standards, with emphasis on addressing all environmental and safety issues. A high-level schedule of the Project has been prepared and is provided in Figure 24.1.

TETRA TECH

Spanish Mountain Gold Ltd.

## Figure 24.1 Schedule of the Project Execution Plan for Spanish Mountain Gold

| ID | Task Name | Duration | Start | Finish |
|---|---|---|---|---|
| 1 | MAJOR MILESTONES | 1057 days | Nov 28 '12 | Dec 17 '16 |
| 2 | Issue PEA | 0 days | Nov 28 '12 | Nov 28 '12 |
| 3 | Issue NI 43-101 | 0 days | Dec 19 '12 | Dec 19 '12 |
| 4 | Start Pre-Feasibility & Feasibility | 0 days | Dec 19 '12 | Dec 19 '12 |
| 5 | Design Frozen | 0 days | May 7 '13 | May 7 '13 |
| 6 | Issue Feasibility Volumes | 0 days | Oct 15 '13 | Oct 15 '13 |
| 7 | Issue NI 43-101 | 0 days | Oct 29 '13 | Oct 29 '13 |
| 8 | Award EPCM | 0 days | Dec 10 '13 | Dec 10 '13 |
| 9 | Start Detail Engineering | 0 days | Dec 11 '13 | Dec 11 '13 |
| 10 | Start Construction | 0 days | Aug 21 '14 | Aug 21 '14 |
| 11 | Start Commissioning | 0 days | Aug 23 '16 | Aug 23 '16 |
| 12 | Start-Up | 0 days | Dec 17 '16 | Dec 17 '16 |
| 13 | PRELIMINARY ECONOMIC ASSESSMENT (PEA) | 95 days | Aug 9 '12 | Dec 19 '12 |
| 16 | ENVIRONMENTAL & PERMITTING | 816 days | Jun 25 '12 | Aug 21 '14 |
| 17 | Environmental Assessment | 721.25 days | Jun 25 '12 | May 22 '14 |
| 40 | Permitting | 242 days | Dec 31 '13 | Aug 21 '14 |
| 58 | PRE FEASIBILITY AND FEASIBILITY STUDY (PFS & FS) | 225 days | Dec 19 '12 | Oct 29 '13 |
| 61 | TRANSMISSION LINES ENGINEERING | 260 days | Apr 12 '13 | Apr 10 '14 |
| 63 | EPCM SELECTION | 88 days | Aug 9 '13 | Dec 10 '13 |
| 67 | DETAIL ENGINEERING | 325 days | Dec 11 '13 | Mar 10 '15 |
| 73 | PROCUREMENT (Long Lead Items) | 953.5 days | Jan 12 '13 | Jul 21 '15 |
| 77 | CONSTRUCTION | 790 days | Aug 21 '14 | Sep 21 '16 |
| 84 | COMMISSIONING AND START-UP | 83.25 days | Aug 23 '16 | Dec 17 '16 |



# 25.0 INTERPRETATION AND CONCLUSIONS

## 25.1 GEOLOGY

SMG has been drilling on the Property since 2005. In total, 670 diamond drillholes (154,368 m) from 2005 to 2012 inclusive have been used in the resource estimate. Deeper drilling from three holes in the Main Zone in 2011 indicates that gold mineralization continues to a depth of at least 480 m.

Based on recent drilling, the geological understanding of the North Zone has increased. It is currently thought that the North Zone Argillite is stratigraphically equivalent to the Upper Argillite unit within the Main Zone. This is significant, since the majority of the disseminated gold in the Main Zone is hosted by the Upper Argillite sequence.

The quality control procedure to monitor possible contamination during the sample collection and preparation comprised the insertion of blank samples into the sample steam. Repeat analysis of blank material sent to ALS within the sample stream gave results within acceptable tolerances, demonstrating no significant contamination during the sample preparation process.

The quality control procedure to measure the precision of the gold values involved the statistical treatment of duplicate pairs for core, reject (prep) and pulp samples. The gold values for the duplicate core and reject (pulp) samples were determined by the metallic gold methods, the same as for the regular samples.

For core samples from the 2012 program, the precision values at the 95% confidence level indicate about a ±21% error for 0.20 g/t gold values, and about a ±49% error for 1.00 g/t gold values. This indicates that higher gold grade samples, which are more likely to contain coarse metallic gold demonstrate a significant nugget effect.

For pulp samples from the 2011 and 2012 programs, the precision values at the 95% confidence level indicate about a ±24% error for 0.20 g/t gold values, a ±12% error for 0.50 g/t gold values and a ±8% error for 1.00 g/t gold values. Note that the pulp samples results comprised the analysis of the -150 mesh material, excluding any coarse metallic gold.

For reject (prep) samples from the 2012 program, the precision values lie, as expected, between those of the core and pulp duplicates. At the 95% confidence level, the precision values indicate about a ±16% error for 0.20 g/t gold values, about a 14% error for 0.50 g/t gold, and about a ±13% error for 1.00 g/t gold values.

The quality assurance procedures to monitor the accuracy of the results comprised reviewing the analytical results from the standards and re-assaying when necessary.

The sample security, sample preparation and analytical procedures during the exploration programs by SMG followed accepted industry practice appropriate for the stage of mineral exploration undertaken, and are NI 43-101 compliant.

The resource estimate by Giroux Consultants updates the earlier 2011 resource estimate, also by Giroux Consultants. It also includes silver in the resource estimate.

Giroux Consultants' updated resource estimate contains 29.36 Mt of 0.60 g/t gold and 0.67 g/t silver in the Measured category; 186.87 Mt of 0.44 g/t gold and 0.69 g/t silver in the Indicated category; and 316.74 Mt of 0.33 g/t gold and 0.65 g/t silver in the Inferred category, based on a 0.20 g/t gold cut-off.

## 25.2 Power Supply to Plant Site

Stantec reviewed and explored six power supply options for preliminary design basis, cost estimate, bill of materials and schedule.

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

## 25.3 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 waste and water management concept presented in this study has the potential to be optimized. The surficial geology and geotechnical conditions should be investigated and the design basis and operating criteria should be further refined.

## 25.4 Mineral Processing and Metallurgical Testing

The gold and silver processing facility to treat 40,000 t/d of mined material was designed based on the results of several metallurigical test programs conducted at G&T and SGS. The process design includes a primary crushing stage, and a SAG and ball mill grinding circuit using cyclones for classification and incorporating a gravity concentration circuit. A flotation circuit will produce an upgraded concentrate for further treatment with the flotation tailings discharged to the TSF. A CIL and subsequent carbon elution and electrowinning facility, will recover the gold into a product suitable for smelting bars of doré. An initial gold recovery of 90.3% will be obtained from a feed grade of 0.7 g/t gold in the first three years of production, followed by a slight decrease in feed grade and recovery in the subsequent years of mining.

## 25.5 ENVIRONMENTAL

Environmental studies and the regulatory review of the Project have been initiated. Biophysical and socio-econmic baseline conditions are being captured for use in an eventual EA. From the information available at this time, none of the environmental parameters identified to-date are expected to have a material impact on the ability to extract the mineral resources or reserves.

## 25.6 MINING

This study indicates that the mineral resource is amenable for extraction using a conventional truck and shovel open pit mining method, and is sufficiently large to sustain a LOM of 15 years, at a mill rate of 40,000 t/d.

The open pit will provide 206 Mt of feed to the mill at average metal grades of 0.48 g/t gold and 0.67 g/t silver. The LOM strip ratio is 2.3:1.

This project included Inferred Mineral Resources that are too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that these data will be realized. An in-fill exploration drilling program is currently planned to potentially upgrade the Inferred Resource to Measured and Indicated categories.

Pit and waste dump designs presented in this report are based on information and assumptions that are adequate for a PEA-level study, and should be considered preliminary. Design assumptions such as pit slopes, pit limits, waste dump slopes, and construction methods, need to be verified with more geotechnical evaluations prior to the next level of study. Waste dump locations require condemnation drilling to ensure there is no potential for sterilization of recoverable minerals.

Though this study is a reasonable assessment of the open pit potential on the resource, the mine plan is preliminary in nature and there is no certainty that the operational aspects of the plan will be realized. The accuracy of the estimate for this scoping level study is ±35%. Further detailed designs and mine plans are necessary to confirm the production schedule, mine productivities and cost estimates.

# 26.0 RECOMMENDATIONS

## 26.1 GEOLOGY

Discovery makes the following recommendations for future work:

- **Structural Interpretation**: A comprehensive structural mapping program on the Main and North Zones is recommended, re-examining field observations and drilling data along with previous interpretations. This work should be integrated with a detailed interpretation of the recent airborne electro-magnetic geophysical survey, when the data become available. This work may aid in determining geologic controls on mineralization and in turn, on geological modelling of the deposit.
- **Diamond Drilling**: In order to re-classify the material currently defined as an Inferred Resource, significant additional drilling will be necessary. Additional drillhole data may allow for data in the Inferred category to be re-classified as Indicated; and for Indicated to be re-classified as Measured. The cost to complete a 35,000 m diamond drill program, in addition to a structural interpretation, is estimated at $7 million.

## 26.2 INFRASTRUCTURE

Additional studies for the location of the process plant area will be required. An investigation of the soil conditions needs to be performed in order to simplify the design of the mill building and major equipment foundations.

### 26.2.1 POWER SUPPLY TO PLANT SITE

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

Due to dynamic and continuous updates from BC Hydro's SIS, Stantec recommends that the facility study for power supply be advanced once BC Hydro issues the final SIS report and identifies the POI for the new 230 kV transmission line.

The cost to complete the above work is estimated to be $30 million.

### 26.2.2 WASTE AND WATER MANAGEMENT

The waste and water management concept discussed in this study has the potential for optimization during subsequent studies, as the geotechnical conditions, design basis and operating criteria are further refined. Knight Piésold makes the following recommendations.

- **Optimization of the TSF layout and location:** Knight Piésold recommends optimizing the TSF embankment locations, particularly the location of the south-east embankment, to ensure that the best option from an engineering, economic and environmental perspective is carried forward. This work will be completed during pre-feasibility and feasibility level studies.
- **Surficial geology study and geotechnical site investigation:** Knight Piésold recommends that a complete surficial geology study and a geotechnical site investigation program be completed in the TSF area to identify local borrow sources and to refine the assumptions made for the PEA cost estimate. The surficial geology study would include a desktop study of the area, followed by ground truthing, laboratory test work and analysis to confirm the finding of the desktop study. The site investigation program would include drilling, (geochemical and hydrogeological to support environmental baseline studies), test pitting, seismic surveys, laboratory test work and analysis. The data collected during the surficial geology study and geotechnical site investigation is required for completion of a feasibility level study. This work is anticipated to cost in the order of $1,200,000.
- **Acquisition of additional property:** Knight Piésold recommends that SMG investigates acquiring some small portions of additional property within the Cedar Creek watershed to ensure that the entire waste and water management footprint is located on the Spanish Mountain claim. Acquisition of additional property is required to complete the geotechnical site investigation program and feasibility level design.

## 26.3 MINING

MMTS recommends that SMG evaluates the offset requirements of the pit from Spanish Creek to an engineering level that matches the next level of study. The design criteria for this off-set will be an important consideration in the design of the ultimate pit, as the selected location of the pit crest will impact the recoverable reserves. The impact of mining activities on Spanish Creek flows and wildlife access needs to be assessed. Assessments will require expertise from hydrogeologists and wildlife biologists, and SMG should seek out those specialists to determine what the studies will cost.

MMTS recommends the development of a plan to determine how and where groundwater that is extracted from depressurization wells will be discharged. For this plan, it is assumed that the discharged water will be suitable for use in material processing. Any deviations from this assumption will affect the operating costs. Though the costs are mining related, MMTS relied on other specialists for the designs and plans for water management.

MMTS also recommends conducting geostatistical analyses to assess the degree of internal dilution in the resource model, and whether the head metal grades will require adjustments. The analyses typically involve creating an adjusted nearest-neighbour model to simulate the de-clustered composite data. Grade-tonnage curves are generated and compared to the actual resource model to determine whether differences are identifiable as dilution. It is estimated that this study would take 50 manhours and will cost approximately $10,000.

The current plant site location limits the capacity of the North Dump. The North Dump is closer to most of the pit than its alternative site, the South Dump. Disposing more waste material at the lower lift elevations of the North Dump will reduce the mine haulage costs. Relocation of the plant facilities further west should be studied by the processing and infrastructure personnel.

The Phoenix Zone should be further explored to either verify or reject the resource potential so that the area may be available for potential pit waste disposal or plant site facilities. An expanded exploration drill program is necessary to determine whether there is sufficient mineralization.

A source for granular materials required for construction and road surfacing needs to be identified. The quality of pit run material poor and is therefore not viable. Failure to identify and acquire a nearby source could result in high supply costs from a distant location. Surface mapping and an augering program may be sufficient to identify this potential resource.

In order to calculate the reserves and update and advance the pit design at the prefeasibility study and feasibility study stages, MMTS estimates costs of $350,000 and $450,000 respectively.

## 26.4 Mineral Processing and Metallurgical Testing

A review of the gravity concentration circuit design is required in order to confirm the number of centrifugal concentrators required for the optimal recovery of gravity recoverable gold. As part of this review, the concept of recovering the gravity gold in its own dedicated electrowinning circuit after the intensive cyanidation step, or combining this gold-bearing solution with that of the main process in one common electrowinning circuit, can also be undertaken.

Additional confirmatory metallurgical test work is recommended prior to conducting the next phase of the Project and include the following aspects:

- Since the reagent CMC is costly, optimization of its addition rate, and the points of addition for maximum effectiveness, is recommended.
- The effect of mass recovery in the rougher, and the subsequent cleaner flotation stages, needs to be quantified in conjunction with the influence of CMC and PAX addition rates.
- Detoxification tests will be required to define the reagent quantities required, and the retention time requirements for completion of the reaction.

The cost to complete the detoxification test work is estimated to be approximately $20,000, while other aspects of the test work are presently being investigated.

## 26.5 ENVIRONMENTAL

Knight Piésold recommends that baseline biophysical studies, First Nation, and community engagement activities continue as the Project design advances in order to support the ongoing EA 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 $2 million. The EA 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 $2.5 million.

## 26.6 RECOMMENDED COSTS SUMMARY

Table 26.1 summarizes the recommended costs for future work of the Project.

**Table 26.1 Summary of Recommended Costs**

| Item No. | Description | Cost ($ million) |
|---|---|---|
| 1 | Infill drilling and structural interpretation | 7.0 |
| 2 | Environmental and permitting | 4.5 |
| 3 | Feasibility study | 8.0 |
| 4 | G&A | 2.4 |
| | **Total** | **21.9** |

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